MODELLING THE OPTIMUM INTERFACE BETWEEN OPEN PIT AND UNDERGROUND MINING FOR GOLD MINES

Seth Opoku

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.

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DECLARATION

I declare that this thesis is my own unaided work. Where use was made of the work of others, it was 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 at any other university.

Signed

……………………………..
(Seth Opoku)

This……………….day of…………….……….2013

ABSTRACT

The open pit to underground transition problem involves the decision of when, how and at what depth to transition from open pit (OP) to underground (UG). However, the current criteria guiding the process of the OP – UG transition are not well defined and documented as most mines rely on their project feasibility teams' experiences. In addition, the methodologies used to address this problem have been based on deterministic approaches. The deterministic approaches cannot address the practicalities that mining companies face during decision-making, such as uncertainties in the geological models and optimisation parameters, thus rendering deterministic solutions inadequate.

In order to address these shortcomings, this research reviewed the OP – UG transition problem from a stochastic or probabilistic perspective. To address the uncertainties in the geological models, simulated models were generated and used. In this study, transition indicators used for the OP - UG transition were Net Present Value (NPV), ratio of price to cost per ounce of gold, stripping ratio, processed ounces and average grade at the run of mine pad. These indicators were used to compare four individual case study mines; with AngloGold Ashanti's Sunrise Dam Gold Mine in Australia, which made the OP – UG transition in 2004 and hence develop an OP – UG transition model. Sunrise Dam Gold Mine is a suitable mine for providing baseline values because it recently made the OP-UG transition. Only four case study mines were used because it took nine months to generate transition indicators for each case study mine.

A generic model was developed from the results of the four case studies to help mining companies make the OP - UG transition decision. The model uses a set of transition indicators that trigger the decision while recognising the uncertainties in the geological models, future mineral price as well as cost and processing parameters. From the generic model, mines can transition when the margin (gold price to cost per ounce ratio) is greater than 2.0; grade is between 4 g/t and 9 g/t, stripping ratio between 3 and 15 m^3/t and positive NPV depending on the type of deposit. With this model mines can now transition when the critical conditions of the transition indicators (gold price to cost per ounce, grade and stripping ratio) are achieved. The model also uses the set of transition indicators to model the probabilistic nature of the OP-UG interface. The derived generic model will help mining companies in their annual reviews to assess the OP - UG interface and make decisions early enough with regard to transition timing.

PUBLISHED WORK

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

- Opoku, S and Musingwini, C. (2012), Modelling geological uncertainty for open-pit to underground transition in gold mines, in *Proceedings of the 21st International Symposium on Mine Planning and Equipment Selection (MPES 2012),* 28th–30th November 2012, New Delhi, India.pp. 503-512.
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TABLE OF CONTENTS

DECLARATION i

ABSTRACT ii

PUBLISHED WORK iii

ACKNOWLEDGEMENTS iv

TABLE OF CONTENTS vi

ABBREVIATIONS xviii

1.0 INTRODUCTION 1

1.1 Background information 4

1.1.1 Status of some open pit to underground transition mines 5

1.1.2 Related research and choice of gold mines as case studies 6

1.2 Research question 7

1.3 Statement of objectives of the thesis 8

1.4 Research methodology 8

1.5 Problem formulation 8

1.6 Thesis structure 9

2.0 REVIEW OF OPEN PIT TO UNDERGROUND TRANSITION 11

2.1 Previous research on open pit to underground transition 12

2.1.1 Cost and stripping ratio 12

2.1.2 Transition depth and its determination 14

2.1.3 Geotechnical challenges 17

2.1.4 Going underground 19

2.1.5 Evaluation of technical and economic criteria involved in changing from surface to underground mining 21

2.1.6 Underground mining: a challenge to established open pit operations 24

2.2 Using Whittle software to determine when to go underground 24

2.3. OP –UG transition framework 25

2.4. Checklist for OP-UG transition 26

2.4.1 Geology 27

2.4.2 Operational 27
2.4.3 Geotechnical 28
2.5 Chapter summary 29
3.0 PROCESS FOR MODELLING OPEN PIT TO UNDERGROUND TRANSITION .30
3.1 Processes followed in the creation of OP-UG transition model 30
3.1.1 Model preparation for simulation 31
3.2 Direct block simulation 34
3.2.1 Quality control and simulation data coverage 36
3.3 Methodology for creating simulated models 36
3.3.1 Problems encountered in simulated models creation 38
3.4 Preparation of simulated models for pit optimisation 39
3.5 Optimisation of open pit and underground mining 40
3.6 Mineable Reserve Optimiser processes 44
3.7 Scheduling using XPAC software 44
3.8 The validity of software used for OP-UG transition 46
3.9 Conceptual OP-UG transition model 47
3.10 Guide for OP-UG transition for gold mines and how to incorporate geological uncertainty in the transition 48
3.11 Chapter summary 49
4.0 DESCRIPTION OF CASE STUDIES: OPEN PIT - UNDERGROUND TRANSITION 50
4.1 CASE STUDY 1: GEITA GOLD MINE 50
4.1.1 Location and background 50
4.1.2 History of Geita Gold Mine 52
4.1.3 Geology and ore body properties 53
4.1.4 Transition plans 56
4.2 CASE STUDY 2: CERRO VANGUARDIA SA MINE 57
4.2.1 Location and background 58
4.2.2 History of CVSA Mine 61
4.2.3 CVSA geology and transition plans 61

4.3 CASE STUDY 3: SADIOLA GOLD MINE......62

4.3.1 Location and background......62

4.3.2 History......63

4.3.3 Geology, current plans and production......64

4.3.4 Transition plans......66

4.4 CASE STUDY 4: MORILA GOLD MINE......66

4.4.1 Location and background......67

4.4.2 History......68

4.4.3 Geology, current plans and production......68

4.4.4 Transition plans......70

4.5 CHAPTER SUMMARY......72

5.0 BASELINE VALUES FOR MODEL USING SUNRISE DAM GOLD MINE AS BENCHMARK......73

5.1 Location and background......73

5.2 History......74

5.3 The use of SDGM for benchmarking against mining industry......75

5.4 Analysis and interpretation of the results......76

5.5 OP - UG transition model baseline results......91

5.6 Chapter summary......101

6.0 CONCLUSIONS AND RECOMMENDATIONS......102

6.1 Conclusions......102

6.2 Research contribution and limitations......102

6.3 Recommendations for future research......103

7.0 REFERENCES......104

APPENDICES......109

Appendix 1: Fields in geological block models......109

Geita......109

Sadiola......109

Morila......112

Appendix 2: Model checking and preparation macros....................................113

Macros for checking models before simulation 113

Macro to prepare models after simulation..113

Macros for Whittle inputs preparation 117

Appendix 3: Grade- tonnage curve data for Morila and CVSA Mines....................123

Morila Mine..123

CVSA Mine...125

Appendix 4: Steps in generating simulated models 127

Importing Datamine drill hole in Isatis 127

Importing Datamine block model 127

Creating intervals for selection..128

Creating Isatis grid...128

Migration of Datamine parameters to Isatis grid......................................129

Creating selection with Isatis grid 130

Data analysis (map, histogram) 131

Exploratory data analysis-point anamorphosis..132

Variogram regularization..133

Block variogram fitting 134

Block Gaussian support correction 135

Direct Block simulation 135

Exporting simulated model to Datamine format..136

Appendix 5: MRO Datamine script and input parameters 137

Appendix 6: Transition evaluation summary 138

Sadiola 138

CVSA 139

Morila 139

Geita..140

Appendix 7: Macros used after optimisation 141

Macros for adding Whittle pit shells 141

Macros for converting block models to wireframes 150

Macros for creating pushbacks 152

Macros to create input models for evaluation (XPAC) 152

XPAC preparation macros 153

Appendix 8: Cumulative distribution data for Geita, Sadiola and Morila Mines.......160

Cumulative distribution processed ounces data for case study mines for Options 1 to 3 160

Cumulative distribution Grade data for case study mines for Options 1 to 3 161

Cumulative distribution NPV data for case study mines for Options 1 to 3.........163

Cumulative distribution Gold price to cost for case study mines for Options 1 to 3 164

Cumulative distribution Stripping ratio data for case study mines for Options 1 to 3 166

Appendix 9: Statistical summary for Sadiola transition indicators..........................169

Sadiola stripping ratio for Option 1...169

Sadiola recovered gold for Option 1...169

Sadiola recovered gold for Option 3...170

Sadiola recovered grade for Option 1 170

Sadiola recovered grade for Option 2 171

Sadiola recovered grade for Option 2 Bi-modal option 1...................................171

Sadiola recovered grade for Option 2 Bi-modal option 2...................................172

Sadiola recovered grade for Option 3 172

Sadiola NPV for Option 1...173

Sadiola NPV for Option 2...173

Sadiola NPV for Option 2 Bi-modal option 1 174

Sadiola NPV for Option 2 Bi-modal option 2 174

Sadiola NPV for Option 3...175

Sadiola gold price to cost per ounce for Option 1..175

Sadiola gold price to cost per ounce for Option 2..176

Sadiola gold price to cost per ounce for Option 2 Bi-modal option 1 176

Sadiola gold price to cost per ounce for Option 2 Bi-modal option 1 177

Sadiola gold price over cost per ounce for Option 3...177

Appendix 10: Statistical summary for CVSA transition indicators..........................178

CVSA stripping ratio for Option 1...178

CVSA recovered gold for Option 1...178

CVSA recovered gold for Option 2...179

CVSA recovered gold for Option 3...179

CVSA recovered grade for Option 1 180

CVSA recovered grade for Option 2 180

CVSA recovered grade for Option 3 181

CVSA NPV for Option 1..181

CVSA NPV for Option 2..182

CVSA NPV for Option 3..182

CVSA gold price to cost per ounce for Option 1...183

CVSA gold price cost per ounce for Option 2...183

CVSA gold price to cost per ounce for Option 3...184

Appendix 11: Statistical summary for Geita transition indicators...........................185

Geita stripping ratio for Option 1 185

Geita stripping ratio for Option 1 Bi-modal option 1..185

Geita stripping ratio for Option 1 Bi-modal option 2..186

Geita recovered gold for Option 1 186

Geita recovered gold for Option 1 Bi-modal option 1 187

Geita recovered gold for Option 1 Bi-modal option 2 187

Geita recovered gold for Option 2 188

Geita recovered gold for Option 2 Bi-modal option 1 188

Geita recovered gold for Option 2 Bi-modal option 2 189

Geita recovered gold for Option 3 189

Geita recovered grade for Option 1...190

Geita recovered grade for Option 1 Bi-modal option 1 190

Geita recovered grade for Option 1 Bi-modal option 2 191
Geita recovered grade for Option 2...191
Geita recovered grade for Option 2 Bi-modal option 1 192
Geita recovered grade for Option 2 Bi-modal option 2 192
Geita recovered grade for Option 3...193
Geita NPV for Option 1 193
Geita NPV for Option 2 194
Geita NPV for Option 2 Bi-modal option 1...194
Geita NPV for Option 2 Bi-modal option 2...195
Geita NPV for Option 3 195
Geita gold price to cost per ounce for Option 1 196
Geita gold price to cost per ounce for Option 1 Bi-modal option 1 196
Geita gold price to cost per ounce for Option 1 Bi-modal option 2 197
Geita gold price to cost per ounce for Option 2 197
Geita gold price to cost per ounce for Option 2 Bi-modal option 1 198
Geita gold price to cost per ounce for Option 2 Bi-modal option 2 198
Geita gold price over cost per ounce for Option 3 199
Appendix 12: Statistical summary for Morila transition indicators...........................200
Morila stripping ratio for Option 1...200
Morila recovered gold for Option 1...200
Morila recovered gold for Option 2...201
Morila recovered gold for Option 3...201
Morila recovered grade for Option 1 202
Morila recovered grade for Option 2 202
Morila recovered grade for Option 3 203
Morila NPV for Option 1...203
Morila NPV for Option 2...204
Morila NPV for Option 3...204
Morila gold price over cost per ounce for Option 1...205

Morila gold price over cost per ounce for Option 2 205

Morila gold price over cost per ounce for Option 3 206

LIST OF FIGURES

FIGURE **PAGE**

Figure 1-1: 3D view of open pit to underground transition (Courtesy: AngloGold Ashanti Limited) 1
Figure 1-2: Layout of thesis structure 10
Figure 2-1: Transition depth [Bakhtavar *et al* (2008)]........15
Figure 2-2: Surface crown pillar developed in a transition from open pit to underground cave mining (Flores, 2004)........18
Figure 2-3: Geological sections for an ideal OP-UG transition (Kurppa and Erkkila, 1967)........23
Figure 2-4: OP – UG transition framework........26
Figure 3-1: Grade tonnage curve of Morila ore body with UC and kriged models 32
Figure 3-2: Grade tonnage curve of Morila ore body showing the simulated AU values 32
Figure 3-3: Grade tonnage curve of CVSA ore body with kriged model........33
Figure 3-4: Grade tonnage curve of CVSA ore body showing the simulated AU values 33
Figure 3-5: Workflow process for uniform conditioning and direct block simulation......35
Figure 3-6: Gaussian point variogram window........37
Figure 3-7: Point variogram fitting window........37
Figure 3-8 : Variogram validation window........38
Figure 3-9: Workflow for creation of simulated models with the direct block simulation method (Source: Geovariances)........38
Figure 3-10 : Flowchart of model preparation macros........40
Figure 3-11: Whittle underground processing option window........43
Figure 3-12: Flow diagram of macros used to prepare models after optimisation 43
Figure 4-1: Map showing the location of Geita Gold Mine (Courtesy: GGM)........50
Figure 4-2: Location of Nyankanga pit and mine infrastructure at Geita Mine (Courtesy: GGM) 51
Figure 4-3: Nyankanga stripping ratio (Courtesy: GGM) 51
Figure 4-4: Nyankanga ore body showing mineralised zones (Courtesy: GGM) 55
Figure 4-5: Geita domains subdivisions (Courtesy: GGM) 55
Figure 4-6: Geita structural trends (Courtesy: GGM) 55
Figure 4-7: Map showing the location of CVSA Mine (Courtesy: AGA) 58
Figure 4-8: Location of CVSA Mine relative to nearby towns (www:argentina.gov.ar)..59

Figure 4-9: Site plan of CVSA (Courtesy: AGA)..59
Figure 4-10: Stripping ratio variation for CVSA...60
Figure 4-11: 3D view of CVSA model for the project (Courtesy: CVSA Mine).............62
Figure 4-12: Map showing location of Sadiola Mine (Courtesy: AGA).........................63
Figure 4-13: W-E section showing the various material types (Courtesy: AGA)...........65
Figure 4-14: Map showing the location of Morila mine (Courtesy: AGA)......................67
Figure 4-15: Section through Morila ore body (Courtesy: Morila Gold Mine)................69
Figure 4-16: 3-D ore body with pit design (Courtesy: AGA) 71
Figure 5-1: Location of Sunrise Dam Gold Mine (Courtesy: AGA) 73
Figure 5-2: Gold mining margins from 2001-2011 (Source: Wright, 2012)...................76
Figure 5-3: Properties of normal distribution...78
Figure 5-4: Cumulative distribution for CVSA stripping ratio for Option 1.....................78
Figure 5-5: Cumulative distribution for CVSA grade for the three Options 79
Figure 5-6: Cumulative distribution for CVSA processed ounces for the three Options79
Figure 5-7: Cumulative distribution for CVSA NPV for the three Options 80
Figure 5-8: Cumulative distribution for CVSA gold price per cost for the three options 80
Figure 5-9: Cumulative distribution for Geita stripping ratio for Option 1......................81
Figure 5-10: Cumulative distribution for Geita grade for Options 1 to 3.......................81
Figure 5-11: Cumulative distribution for Geita processed ounces for Options 1 to 3....82
Figure 5-12: Cumulative distribution for Geita NPV for Options 1 to 3 82
Figure 5-13: Cumulative distribution for Geita Gold price to cost for Options 1 to 3 83
Figure 5-14: Cumulative distribution for Sadiola stripping ratio for Option....................83
Figure 5-15: Cumulative distribution for Sadiola grade for Options 1 to 3 84
Figure 5-16: Cumulative distribution for Sadiola processed ounces for Options 1 to 3.84
Figure 5-17: Cumulative distribution for Sadiola NPV for Options 1 to 3......................85
Figure 5-18: Cumulative distribution for Sadiola gold price to cost for Options 1 to 3...85
Figure 5-19: Cumulative distribution for Morila stripping ratio for Option 1..................86
Figure 5-20: Cumulative distribution for Morila grade for Options 1 to 3 86
Figure 5-21: Cumulative distribution for Morila processed ounces for Options 1 to 3...87
Figure 5-22: Cumulative distribution for Morila NPV for Options 1 to 3 87
Figure 5-23: Cumulative distribution for Morila gold price to cost for Options 1 to 3.....88
Figure 5-24: Sectional view of CVSA showing pit outlines..89
Figure 5-25: Sadiola recovered gold for Option 2 with Bi-modal distribution 89
Figure 5-26 : Sadiola recovered gold for Option 2 with Bi-modal Option 1..................90
Figure 5-27 : Sadiola recovered gold for Option 2 Bi-modal Option 2.........................90
Figure 5-28: OP – UG transition model flowchart 100

LIST OF TABLES

TABLE..**PAGE**

Table 1-1: Status of some open pit to underground transition mines 6

Table 3-1: OP-UG transition interface calculations 42

Table 3-2: Mining calendar for Geita Option 1 45

Table 3-3: Mining calendar for Geita Option 2 46

Table 4-1: Open pit mining fleet for Geita Mine 52

Table 4-2: Geita mining statistics...52

Table 4-3: Chronological list of events at Geita Gold Mine (GGM)..............................53

Table 4-4: Project level classification (Courtesy: AngloGold Ashanti Limited)..............57

Table 4-5: CVSA mining fleet 60

Table 4-6: Chronological list of events at CVSA Mine...61

Table 4-7: List of chronological events at Sadiola Mine 64

Table 4-8: Sadiola mining fleet 66

Table 4-9: List of events at Morila Gold Mine...68

Table 4-10: Morila Mining fleet 70

Table 4-11: Summary of rock types in the resource model 70

Table 4-12: Mining production statistics...70

Table 5-1: Chronological list of events at Sunrise Dam Gold Mine 74

Table 5-2: SDGM 1996-2002 production summary...75

Table 5-3: Comparisons of mine statistics for the case study mines (Courtesy: AngloGold Ashanti) 77

Table 5-4: Key transition indicators for base model 92

Table 5-5: Key transition indicators for simulated models 92

Table 5-6: Summary of weighted transition indicators for base model 94

Table 5-7: Summary of weighted transition indicators for simulated model..................94

Table 5-8: Similarities and differences between Sunrise Dam Gold Mine and the case study mines...96

Table 5-9: Transition indicator values at 95% cumulative probability 97

Table 5-10: Transition indicator values at 90% cumulative probability 97

Table 5-11: Sensitivities of the transition indicators to gold price for Option 1.............98

Table 5-12: OP-UG transition indicators in relation to baseline values.........................99

LIST OF UNIT SYMBOLS

Billion Years .... Ga
Million Ounces .... Moz
Million Tonnes .... Mt
Ounces .... oz
Thousand Ounces .... koz
US Dollar .... USD
Australian Dollar .... A$
Billion .... bn

ABBREVIATIONS

AGA .... AngloGold Ashanti Limited
BCM .... Bank cubic metre
BIF .... Branded Iron Formation
BIFC .... Chemical BIF
BIFS .... Sedimentary BIF
BUP .... Business Planning
CAR .... Continental Africa Region
COV .... Coefficient of variation
DBSIM .... Direct-block conditional simulation
EW .... Equal Weighted
GMP .... General Mining Package
H_{td} .... Transition depth
H_{tp} .... Transition point
IRR .... Internal Rate of Return
KPI .... Key Performance Indicators
LoM .... Life of Mine
MCF .... Mine Call Factor
MRO .... Mineable Reserve Optimiser
NPU .... Near Pit Underground
NPV .... Net Present Value
NQ .... Diamond drill core of 53mm diameter
OP - UG .... Open_pit to Underground
OP .... Open_pit
ORD .... Ore Reserve Development
RUC .... Reverse Circulation Drilling
SD .... Standard Deviation
SGS .... Sequential Gaussian Simulation
SIBC .... Stay in Business Capital
SMU .... Smallest Mining Unit
SR .... Stripping ratio
T_L .... Transition level
UC .... Uniform Conditioning
UFS .... Underground Feasibility Study
UG .... Underground

1.0 INTRODUCTION

Open pit mining is generally considered to be more advantageous as compared to underground mining due to its mass production and minimum cost. If an ore body is large and extends from surface to “great depth”, the part of the deposit close to the surface is usually mined from an open pit to give early revenue while preparations are being made for mining the deeper parts by underground means. Many surface mines are increasingly becoming aware of the value gained by considering underground options early in the open pit mining life. The choice of mining method and open-pit limit for a specific mineral deposit depends on factors such as the geological conditions of the ore body, stripping ratio, extraction depth and economic, community, social and environmental requirements. If a deposit changes much in geometry along the strike, especially if the change occurs at the ends of the deposit as in Figure 1-1, the stripping ratio will be too large when the whole deposit is mined by open-pit mining even if a pushback is considered. In this case, it is more suitable to have the deposit mined by combined mining methods to maximise the return on the investment. The problem is when and where to fit the underground production schedule to the open pit to maximise its value.

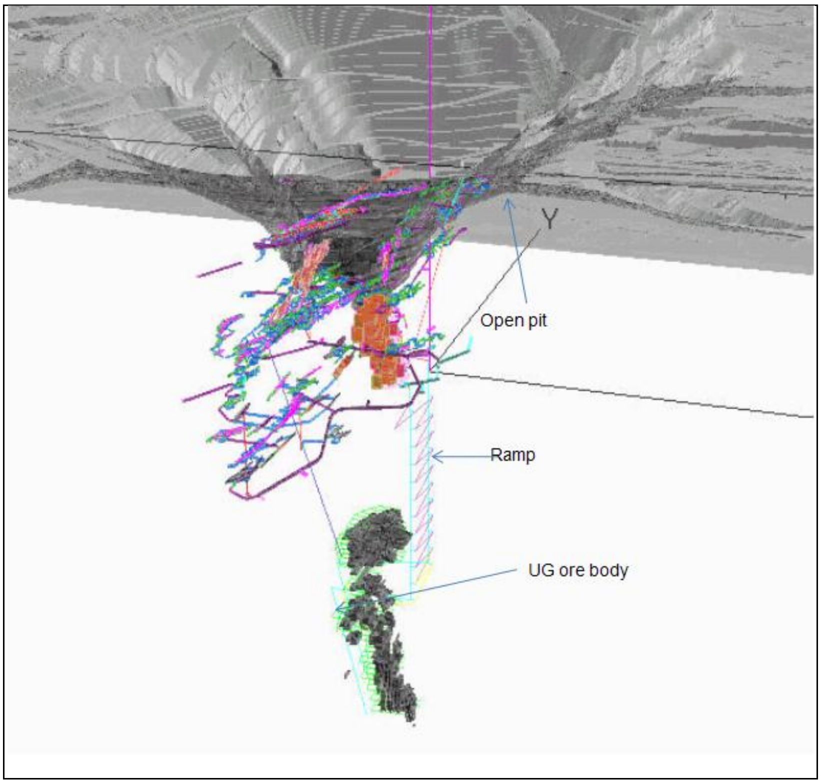

Figure 1-1: 3D view of open pit to underground transition (Courtesy: AngloGold Ashanti Limited)

There are three basic ways of analysing the possibility of accomplishing the open pit to underground (OP– UG) transition. These are from open pit (OP) to underground (UG), from underground to open pit or having both mining methods working simultaneously to extend the production life or to increase production. If the combined method is chosen, there should be successful interaction between the open pit and the underground methods in order to supply a continuous flow of ore to the plant. There are various known challenges during OP-UG transition; however, the methodologies available to address the problem have been based on deterministic approaches. The deterministic approaches fail to take into account the uncertain nature of the parameters used during optimisation as well as the geological uncertainties and hence fail to address the real transition problem. There is the need for a well-structured approach in solving the timing for OP-UG transition to maximise Net Present Value (NPV), which is one of the key financial indicators used during mining project feasibility studies to minimise risks. Most mines would consider an underground option or the combined approach only when the open pit fails to yield the expected results, or the pit is nearing its completion due to lack of a transition model to use in the decision making process. It is a common phenomenon to determine an optimum interface between the open pit and underground mining in conjunction and run alternative scenarios on the open pit in an effort to delay future waste stripping costs. The transition depth (level) is one of the numerous factors that dictate the change of mining method from open pit to underground. The problem with open pit to underground transition involves the decision of when, how and at what depth to transition from open pit to underground. Current criteria for OP–UG transition are not well defined and documented, as most mines rely on the experiences from their project feasibility teams.

There are many factors taken into consideration when underground mining becomes more profitable than the open pit mine. One of the major factors seen in trying to evaluate what the best interface between the two would be is the lack of required information (geological and bankable feasibility documents) being available early enough in the OP–UG transition. Most mines, when faced with the question as to when they should go from an open pit to underground, lack the necessary information to make that decision. The mines have a vague idea about the geology and the ore body value below the open pit. An information gathering stage could be initiated which will typically start as a diamond drilling exercise often followed by sinking an exploratory shaft or winze to augment the diamond drilling information. Ground conditions, pit depth, and factor of safety can have a large impact; there are many factors involved in this decision, strategically and financially. There is often a point where a decision has to be made whether to continue deepening the mine or changing to underground methods. The question most mines face is why one has to evaluate an underground mining method in an early stage of the open pit planning or during open pit production. However, the reality is that the economic final pit is usually closer than one thinks. Perhaps one of the most important decisions, in the initial stages of a project for a transition from open pit to underground mining, is the definition of the most suitable underground mining method based on the characteristics of the deposit and, at the same time, the economic and business requirements of the mining company. Business requires high production rates and low operational costs. In choosing between OP and UG, the time to transition is critical to maximise the value of the resource. A well-balanced schedule needs to be maintained during the transition period to maintain a constant production profile.

There are opportunities to add value if the timing of the underground and open pit mining fits appropriately into the company's strategic plan. Proper planning during open pit to underground transition is done with the aim of optimising mineral production and rationalising waste stripping to manage the stripping ratio, and thereby reducing the operating and capital costs. In addition, mining fleet rationalisation during the transition is prepared to ensure the best fit-for-purpose and cost effective fleet to be utilised. In making a choice between OP and UG, the time to transition is vital to maximise the value of the resource and to keep the window of opportunity opened. Comprehensive budgeting, anchored on good operating, capital cost estimates and proper scheduling of the expenditures, and timely execution of plans in each department and section are necessary in order to achieve a good transition. In mining, the capacity determines the rate of extraction and hence exhaustion of the reserve. Thus, there is usually an interaction between the capacity decision and the production decisions. An underground mine requires large up-front capital in the form of shaft access, development and equipment, and the cost can be in the order of billions of United States Dollars (USD). Obtaining approval for this kind of money requires comprehensive information to justify a big upfront spend of capital.

To deepen an open pit beyond its ultimate depth is expensive and time consuming given that the stripping ratio will change as the pit deepens. Every mine and its deposit are unique but there are common factors such as those encountered in diamond pipes. The underground mine will normally be directly underneath the open pit whereas in copper, gold and other deposits there might be enough space available to locate the underground mine away from the open pit. Many factors affect the decision on whether to commence a mine as an open pit (OP), underground (UG) or transition from open pit to underground (OP-UG) at a later stage. Some of the impacts include the cost of stripping which may reduce cash flow from the operation and may not be strategically desirable at the beginning of the project.

Vertical narrow vein ore bodies are more suitable to underground mining, whereas open pits have high stripping ratios. On the other extreme, large porphyry ore bodies can be mined at a very low cost by open pit if they are close to the surface. For complex ore bodies (geology not well understood), open pit resource recovery can make the open pit cheaper than underground mining as generally all rock within the pit shell is mined. In an open pit, the ore can be separated through the grade control process, but in an underground mining scenario, mining costs are higher and the goal is to minimise waste mining – this means that smaller areas of ore that can be mined by open pit may be left in an underground scenario, usually as pillars. In UG mining, the mining is not done from top down (as in a pit) and would be more selective. If the mine is mill constrained, there is an opportunity to “high grade” the mine at the beginning of the mine life to allow higher cash flows. If resource recovery is not a requirement then the cut-off grade is lifted in order to lift the head grade. An open pit exposes the ore whereas in underground mining it is easier to limit access. When a pit goes deeper the stripping ratio increases, mining costs escalate with depth, haulage distances increase, wear and tear on the equipment (truck tyres) increases. The rock conditions change as the pit gets deeper resulting in tighter blast patterns, which increase blasting costs, more groundwater, and surface water needs to be pumped out, profit margin begins to decline and the incremental value of the pit gets smaller (www.gemcomsoftware.com).

1.1 Background information

Many open pit mines are planning or implementing the process of open pit to underground transition and many of them have encountered problems during the implementation stage of the transition processes after feasibility studies and have not been able to follow their feasibility plans to the end. Some of these mines include Palabora, Finsch and Venetia in South Africa; Bingham Canyon in the USA; Chuquicamata and Mansa Mina in Chile; Grasberg in Indonesia; Kidd Creek Mine, Doyon Gold Mine, and Dome Mine in Canada; Jwaneng Mine in Botswana; Telfer, Argyle, Mount Keith and Sunrise Dam in Australia and Geita Mine in Tanzania. Some OP-UG transition problems include instability in the areas closer to the underground operation, deteriorating haulage roads, increasing probability of slope failures and unsafe working conditions and underground flooding due to the groundwater and/or surface water inflow.

1.1.1 Status of some open pit to underground transition mines

A mine can be a surface (strip mine or an open pit mine) or an underground mine. The mining method used depends on the depth, lateral extent and economic value of the ore being mined. The deepest underground mine is in West Wits (about 3.5 km), a South African gold mine, while an open-pit Bingham Canyon Mine is more than 4 km wide and more than 1 km deep. The shape of a mineral deposit (small, irregular, deeply buried, narrow, vein) mostly dictates the choice between open pit and underground mining. In 1955, Butte mine in the USA began the transition from underground to open pit. Butte mine is among few mines to transition from underground to open pit. Palabora mine in South Africa had to transition from open pit to underground using the block caving mining method when the pit reached 800 m depth. Doyon Mine began the open pit mine in 1980 and commenced the underground in 1985. Table 1-1 shows a status of some open pit to underground transition mines.

Table 1-1: Status of some open pit to underground transition mines

Mine	Transition Year (Actual / Planned)	Reference
Argyle diamond mine	2005	Bull *et al* (2004); Hersant (2004)
Bingham Canyon	2014	Flores (2004)
Bronzewing	1991	Luxford (1997)
Chuquicamata	2018	Arancibia and Flores (2004)
Darlot	2008	Luxford (1997)
Diavik	2010	idexonline.com
Ekati Diamond mines	2006	Jakubec (2004)
Geita Mine	2013	AGA reports
Grasberg copper-gold mine in Indonesia	2016	Brannon (2004); Srikant *et al (*2007).
Jundee	1997	Luxford (1997)
Kanowna Belle gold mine in Australia	2008	Kandiah (2007)
Kiruna mine	1999	Kuchta *et al (*2003)
Mt McClure	1994	Luxford (1997)
Palabora mine	2004	Brummer *et al (*2006)
Scuddles Mine	2016	Luxford (1997)
Sunrise dam mine	2003	AGA reports
Telfer in Australia	2002	Arancibia and Flores (2004)
Tulawaka, Tanzania	2005	Barrick reports
Venetia diamond mine	1992	Flores (2004)
Wiluna	2009	Luxford (1997)
Woodlawn	1980	Luxford (1997)

1.1.2 Related research and choice of gold mines as case studies

Most researchers have used the breakeven cut-off grade criterion to define ore as a material that will just pay mining and processing costs. This criterion is not optimal since it only separates the ore from the waste but the mine planner often seeks to optimise the cut-off grade of ore to maximise the NPV. The determination of the optimum cut-off grade for a single metal deposit can be very complex even when price and cost are assumed to be constant. This is because it involves the costs and capacities of the several stages of the mining operations, the waste/ore ratios and average grades of different increments of the ore body. If mineralisation extends beyond a certain depth from the surface of a pit, the stripping ratio (SR) becomes too high. It should then be converted to an UG mine. Optimisation of the transition problem was, and still is, an important issue in mining.

This thesis modelled the open-pit to underground transition problem, which was researched from a stochastic or probabilistic point of view by developing a model for mining companies to transition decisively and smoothly. Sometimes, mining companies are forced to simplify their operations, and tend to make simple statements of how many tonnes of a certain grade they can produce (and market what they can sell). Therefore, it is common in open pit gold mines to work with fixed stripping ratios, cut-off grades, beneficiation rules and product specifications. The challenge is now for mining organisations to see how quickly they can step up their management processes to see exactly what their resources are actually capable of delivering.

Information gathering for OP-UG transition is a difficult task. The reserves for various companies are generated from resource models. Most mining companies have a confidentiality associated with them making it impossible to obtain geological models. Data from gold mines in only one mining company (AngloGold Ashanti Limited), the researcher's employer, that have had to change their transition plans since there was no model to follow to assess OP-UG transition, were used for the study. The time involved in running a model to generate the transition indicators for each of the four case study mines was about nine months. This time constraint limited this study to four case study mines, although AngloGold Ashanti Limited, one of the world's leading gold mining companies, has 21 operations in 10 countries on four continents.

1.2 Research question

There are few methods available to mining companies in making the decision as to when to transition from open pit to underground. The most common one is by comparing the differences in the financial returns of a pushback, to mining the same by underground means, using optimisation software such as Minemax (global optimiser that seeks to maximise NPV). Most of the studies done on open pit to underground transition were based on the transition depth (H_{td}). However, this is inadequate because as indicated earlier, there are other factors that are critical to the transition decision. These factors change over time and make the transition depth dynamic or uncertain hence the hesitation or delays by mines to make the transition. The question is therefore:

> *"Is there a set of appropriate criteria or indicators that can be utilised to trigger the transition decision from open pit to underground mining given the uncertainties in the geological models, gold price as well as cost and processing recoveries?"*

1.3 Statement of objectives of the thesis

The main objectives of this research were to:

- Identify appropriate transition indicators for open pit to underground transition;
- Develop a stochastic model using transition indicators based on grade or geological uncertainty for the open pit to underground transition. This model will help reduce possible loss of the huge capital investment during OP-UG transition and enhance surface and underground mine planning processes by incorporating more flexibility in the planning process; and
- Test the OP–UG transition model using baseline values.

1.4 Research methodology

The methods employed in this study included:

- Collection, extraction, collation and validation of geotechnical data, mining data, and geological models on the various mine sites;
- Testing of OP – UG transition model using values of transition indicators for Sunrise Dam Gold Mine as baseline values;
- Analytical and statistical evaluation of the results; and
- Comparing the OP – UG results against industry norms.

1.5 Problem formulation

Uncontrollable parameters in OP to UG transition include:

- Gold price;
- Ore body geometry, and

- Infrastructure (location of the mine).

Controllable parameters in OP to UG transition include:

- Mining method;
- Timing (window of opportunity);
- Cost, plant recoveries, mine call factor (MCF);
- Cut-off grade; and
- Stripping ratio.

One way to determine OP-UG transition is to convert the open pit pre-stripping ratio of cost per tonne and compare the value to the underground mining cost. This is the standard financial analysis approach. This is different for each ore body. Transitioning from open pit to underground may change the mine from a mill-constrained scenario to a mining constrained one when mining underground only.

Most transition mines have faced one problem after the transition. Some of the factors contributing to the transition problem are as follows:

- The effect of change in the gold price and cost;
- Ability to maintain the required plant throughput during and after transition;
- Geotechnical challenges and stability of the rock mass;
- Lack of confidence in the geological resource model;
- Environmental factors like subsidence, which sometimes favour open pit rather than underground mining;
- Lack of expertise to make the transition; and
- Capital required to transition.

1.6 Thesis structure

The structure of the thesis is illustrated in Figure 1-2. There are six chapters. Chapter 1 introduces the thesis work which clearly states the research question, the problem definition, objectives, methodologies applied to achieve the objectives, scope of work, as well as the organisation of the report. Chapter 2 reviews the OP - UG transition literature. The modelling process in solving the OP - UG transition problem and the conceptual transition model are explained in Chapter 3. Chapters 4 discuss the four case study mines in this particular order: Geita, Cerro Vangudia SA, Sadiola Gold Mine and Morila Gold Mine. Chapter 5 analyses the Sunrise Dam Gold Mine to provide baseline values for the model and lastly, Chapter 6 has the conclusions and recommendations based on the results of the study.

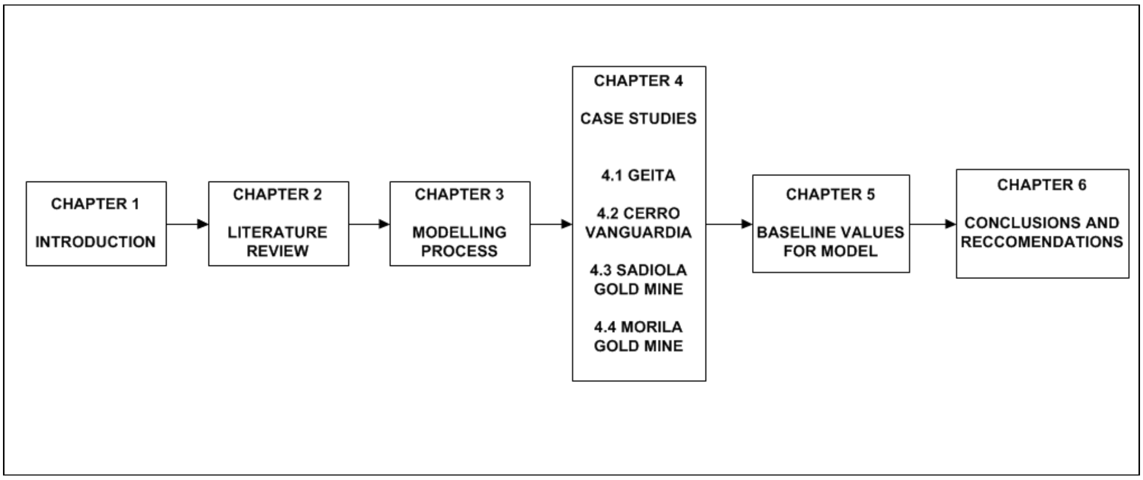

Figure 1-2: Layout of thesis structure

2.0 REVIEW OF OPEN PIT TO UNDERGROUND TRANSITION

Open-pit mining is generally preferred by most mine planners and investors compared to underground mining where an ore body is located close to the surface, large enough and has little overburden. Underground mining options for extraction are considered at a point in the mining life when economic conditions become impossible to continue mining using open pit methods. At this point, comparisons are made between mining the deposit by open pit or underground. The point at which the mining method is changed from open pit to underground is often referred to as the transition point (H_{tp}) or transition depth (H_{td}). The transition point could occur anywhere from pre-feasibility project stage to years after commencement of mining. The transition point at which an underground mine becomes more economic than an open pit operation is not a single evaluation but depends on many factors. It is therefore more reasonable to refer to a transition point rather than transition depth.

For an outcropping ore body, it is best to be mined by open pit down to the point where the cost of mining the last tonne is equal to the cost of mining that tonne from underground. The last cut in the open pit is generally marginal while the first production from underground is the most costly because it will take months to develop the sequence of stoping required to meet full production capacity. One of the most important decisions in the initial stages of a mining project is the choice of a suitable mining method (open pit or underground) based on the characteristics of the deposit and the economic and business requirements of the mining company. If the business requires high production rates and low operational costs, then the method could include an open pit mining or an underground caving mining method.

Factors that affect the ideal transition from OP to UG mining are cut-off grades, waste stripping, portability of skills from surface mining experience to underground mining environment, stockpile generation and reclamation, capital requirements, tailings capacity, closure cost implications, as well as the decision of what depth and when to make the transition. Currently, the criteria for making this transition are not well defined. Some of the factors that can affect OP-UG transition can be listed as follows:

- UG cost (sensitive to depth);
- Time to transition is critical to maximise the value;
- Mining cost (determines when to transition);

- Stripping ratio (required break even to transition);
- Huge capital required for UG project;
- Unit cost of surface mining (increases with depth due to amount of waste to be removed);
- Ore body configuration determines how to transition (size of the deposit and final pit slope angle);
- Types of material found below a certain depth and the availability of the processing methods for treatment or modification on the existing plant constrains UG transition;
- Decisions as to how and when to act including the extraction and routing of blocks of ore, the timing of decisions such as pushback or transitions;
- The placement of shafts; the ratio of ore to waste in OP controls the transition level; Open pit should mine ore bodies whose stripping ratio (SR) does not exceed the break even stripping ratio;
- Conversion of mining equipment from OP to UG mining;
- High risk assessments in OP mining constrained pits to be mined below certain depths and transition is therefore required earlier than anticipated; and
- The depth at which free cash flow becomes negative.

2.1 Previous research on open pit to underground transition

The following section details work done by other authors in trying to address the OP-UG transition decision. Among various parameters considered by the authors were cost, stripping ratio, transition depth, geotechnical challenges and using Gemcom's Whittle 4X software as a tool to assess if going underground is feasible (www.gemcomsoftware.com).

2.1.1 Cost and stripping ratio

Luxford (1997) briefly discussed some of the issues involved in making the transition from open pit to underground mining. His aim was to flag the critical issues when planning to make the transition from open pit to underground mining and to identify critical aspects of mine development. Luxford (1997) discussed the OP-UG transition issues, with emphasis on gold and copper deposits in Australia. He said that many open cast mines were developed on shallow oxide reserves but have exhausted these reserves and these mines have made the transition to the deeper sulphide ore whilst some of the operations have reached a point where decisions will soon have to be made to transition from OP–UG mining. Luxford (1997) made the following points:

- Mining companies use open pit-mining methods if the reserves are in the shallow oxides because the oxide rocks are mostly soft and cannot support underground mining;
- Cost usually drives the decision to take an open pit mine underground. He argued that as open pit stripping cost keeps rising, as the mine gets deeper, there comes a time when underground mining cost will be less than the open pit mining cost. At that point, which this research proposes as transition point (H_{tp}), a decision to choose between extending the open pit mine and going underground is made after considering detailed analysis of all operational and capital costs; and
- Capital costs are often a factor in the choice between a major pushback and going underground. It seems reasonable that cost is one of the many factors that determine the OP-UG transition. Open pit mining should continue until the underground mining cost becomes cheaper than the open pit mining cost before the detailed cost analysis is made.

However, Luxford (1997) did not mention the mining method being used to exploit the sulphide ore neither did he show how the open pit and underground cost could be calculated in making the transition decision. Luxford’s views on the following are still applicable:

- Workforce recruitment;
- Ore body geometry;
- Ore handling;
- Production rate;
- Decline, conveyor or shaft;
- Ventilation, and
- Geomechanics.

2.1.2 Transition depth and its determination

Hayes (1997) discussed the impact, which makes UG mining more economic than OP mining and noted the importance of issues such as management competence and system, geological setting, geotechnical characteristics, stripping ratio, productivity and capital cost in making this decision. He included the following factors in determining the transition depth as: the mineral resource, the mining method and the mining cost factors.

Finch (2012) stated that the OP-UG transition problem manifests itself in two ways. These are sequential and parallel mining. In the sequential mining, the UG mining is directly beneath the open pit, whereas with the parallel mining there is an opportunity to site the underground portion away from the open pit mining allowing for simultaneous mining of both the OP and the UG. Finch (2012) argued that to determine the optimal transition point the following issues need to be evaluated:

- Availability of feed;
- Feed grade;
- Resource utilisation impact;
- Stripping ratio;
- Price;
- Production rate; and
- Mining cost.

Finch (2012) dwelled much on the transition point evaluation so that the point that offers the higher values can be chosen. The point Finch (2012) made was valid however, the transition point cannot be complete until a point in time is determined since the factors involved in its determination change over time.

The model for determining the optimal transition depth from open pit to underground mining by Bakhtavar *et al* (2008) stated that the most significant problem at that time was the determination of optimal Transition Depth (H_{td}) from OP to UG mining. Bakhtavar *et al* (2008) used a heuristic algorithm as a basic model based on Block Economic Value of OP and UG. They derived their formulae based on the allowable and overall stripping ratios. For this objective, an analytical procedure was produced. The contemplated model is about deposits with outcrops or overburden and including maximum or minimum possible pit floor width. About the tabular deposits including outcrops and considering the maximum width of pit floor for exploitation, a simple effectual formula was proved. In the second case, to take into account the eventual deepening of the OP without extending it sideways, instead of maximum width, minimum possible width of pit floor was contemplated. The formulae are based on the ore below a certain thickness of overburden, which relates to the maximum and minimum possible width of pit floor. A general schematic illustration of the transition problem is in Figure 2-1. For a steeply dipping ore body of uniform width, the optimal depth of the open pit is a function of the stripping ratio and ore body continuity, which in turn is a function of the prevailing economic and technological conditions such as the price of the mineral on the world market and the political economic conditions in the country. Figure 2-1 shows the ideal block model used by Bakhtavar *et al* (2008) to derive the transition depth equations.

Bakhtavar *et al* (2008) concluded that selection of mining method is one of the most important decisions in the design stage of a mine and before development. They stated that in relation to the deposits, which have the potential of using the combined mining methods (OP and UG) in the vertical direction, the most significant problem is the H_{td} determination, which could be determined by using Equations 2-1 to 2-4. Equation 2-1 is used if the deposit includes outcrops and maximum width of pit floor. Equation 2-2 is used if the deposit includes outcrops and minimum width of pit floor. Equation 2-3 is used when the deposit includes overburden and maximum width of pit floor, while Equation 2-4 is used when the deposit includes overburden and minimum width of pit floor. Figure 2-1 shows the various parameters used in deriving the transition depth.

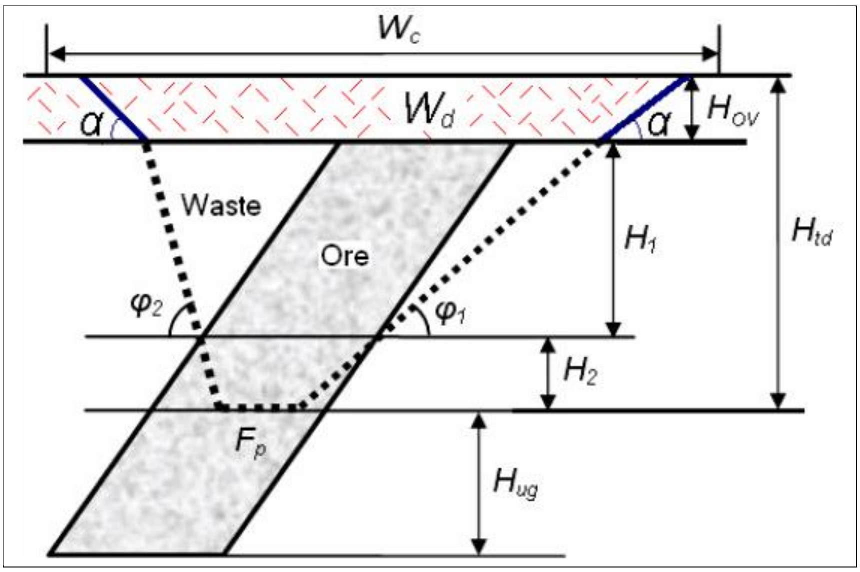

Figure 2-1: Transition depth [Bakhtavar *et al* (2008)]

$$H_{td} = \frac{W_d\left(R_{ug} \times C_{ug} - R_{op} \times C_{op}\right)}{C_w \times A}$$ **Equation 2-1**

$$H_{td} = \frac{W_d\left(R_{ug} \times C_{ug} \times C_{op}\right) + \left(W_d - F_p\right) \times C_w}{C_w \times A}$$ **Equation 2-2**

$$H_{td} = \frac{\left\{W_d\left(R_{ug} \times C_{ug} - R_{op} \times C_{op}\right)\left[2B + A\right]\right\} + \left[C_w \times \left(W_d - F_p\right) \times A\right]}{2C_w \times B \times A}$$ **Equation 2-3**

$$H_{td} = \frac{\left\{W_d\left(R_{ug} \times C_{ug} - R_{op} \times C_{op}\right)\left[2B + A\right]\right\} + \left[C_w \times \left(W_c - W_d\right) \times A\right]}{2C_w \times B \times A}$$ **Equation 2-4**

Where: H_{td} = Transition depth (m).

W_d = Horizontal thickness of the ore body (m).

R_{ug} =Ore recovery coefficient via underground method.

R_{op}=Ore recovery coefficient via open pit method.

Φ_1 = Pit side slope angle along foot wall.

Φ_2 = Pit side slope angle along hanging wall.

A = Cot Φ_1 + Cot Φ_2.

C_{ug} =Full prime cost of 1 ton of the mined mineral via underground.

C_{op} =Prime cost of 1 ton of the mined mineral via open pit.

C_w =Total cost of 1 m^3 of ground removal via open pit mining.

W_c =Length of base of overburden trapezium.

F_p =Minimum possible width of pit floor.

B = Cot α.

Bakhtavar *et al* (2008) used hypothetical cases and not real case studies to derive the transition depth and stated that the significance and usability of Equations 2-1 to 2-4 would be achieved by utilising them to determine the transition depth (H_{td}) of some various practical cases. The equations derived by Bakhtavar *et al* (2008) are static models, yet the transition problem is dynamic, hence H_{td} should be $H_{td\ t}$ where t is the point in time at which prices and costs are obtained or estimated. This is the reason why this research study adopted a stochastic approach in order to capture the dynamic nature of the problem.

2.1.3 Geotechnical challenges

Flores (2004) pointed out some of the geotechnical challenges associated with caving during open pit to UG transition by using Chuquicamata mine as a case study. This was carried out through the International Caving Study Stage II (ICS-II), managed by the Julius Kruttschnitt Mineral Research Centre, Brisbane, Australia, of which CODELCO is one of the sponsors. That study concluded that there is currently neither sufficient experience in transition for deep pits nor available design methodologies in spite of the topic's importance to the mining industry. The only documented transition available at that time involving a large open pit and underground mining by caving was Palabora mine, South Africa (Glazer and Hepworth 2004). Figure 2-2 shows the crown pillar development from the open pit to underground transition at Chuquicamata mine. Some of Flores findings were as follows:

- When the final pit is reached in 2013 with a depth of 1,100 m, the undercut level will be located at a depth of around 1,500 m from surface;
- Cave initiation and propagation. The initial stage of the underground mining will be in a hard and massive rock mass, where cave initiation and propagation may be difficult;
- Simultaneous open pit and underground mine operations. The economic and business requirements of Chuquicamata mine are such that a period of simultaneous open pit and underground mining would be required. Hence, at least for a certain period, a stable crown pillar must be maintained between the cave back and the pit bottom;
- Subsidence, once the caving connects to the pit bottom the pit will become a subsidence crater with a zone of influence extending beyond the pit perimeter; and
- Groundwater, due to the presence of groundwater in the slopes of Chuquicamata's open pit and some rains during the Bolivian winter (January and February), there is a non-zero probability of inrushes of water or mud into the underground mine. These inflows or mud-rushes could be worsened by the presence of major geological structures.

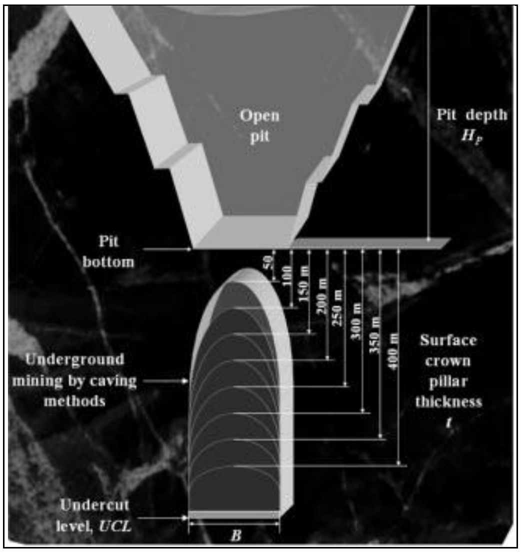

Figure 2-2: Surface crown pillar developed in a transition from open pit to underground cave mining (Flores, 2004)

Flores (2004) stated that the decision to make the transition from open pit to an underground operation is often based on a simple determination of the NPV of the next feasible open pit pushback. Underground mining is only contemplated when a further pushback is shown to be uneconomic and stated that any decision to go underground also requires consideration of a wide range of technical factors, and careful planning. This means a significant amount of time is needed for achieving underground mining and up to 20 years was suggested by Stacey and Terbrugge (2000).

Stacey and Terbrugge (2000) suggested that the transition problem was known but the lack of a model to address the timing remained an issue. In such cases, it is desirable that the open pit continues its operation during the first stages of underground mining, and that the underground mine gets to a high level of productivity quickly before closure of the open pit operation. This means that there will be a period of simultaneous open pit and underground mining operations. Flores (2004) stated that the simultaneity implies an interaction between the open pit and underground mining, which makes the problem more complex than the typical open pit or underground mine designs. The presence of the deep open pit will affect the stress field in which the underground mine will be developed and, conversely, the propagation of the caving will affect the stability of the surface crown pillar that defines the bottom of the open pit. Additionally, many other factors or potential hazards could make the problem even more difficult if these are not identified prior to making the transition from open pit to underground mining.

Stacey and Terbrugge (2000) highlighted that the following aspects must be considered during the OP-UG transition:

- The planning and implementation period for transition from OP-UG could take as long as 20 years. In addition, they suggested that the planning must commence at an early stage, which is indeed true requiring annual reviews of the OP – UG transition decision;
- An economically designed pit, should have slopes that are close to their stability limits and little scope for extending the open pit mining to greater depths, other than with a pushback;
- Surface and underground infrastructure is often at risk due to deepening of pits, underground mining below pits, and deepening of underground mining beyond planned depths;
- Introduces the risk of mud rushes from within the rock mass;
- Air blasts occurs because of underground collapses in association with mud rushes;
- The presence of an abandoned pit above underground workings can lead to greater risks of dilution and mud rushes; and
- The choice of underground mining method has a major effect on the stability of the surface.

Although Stacey and Terbrugge (2000) suggested transition timing up to 20 years they did not provide any transition criteria to guide the OP-UG transition.

2.1.4 Going underground

Fuentes (2004), in his paper on going to an underground (UG) mining method, stated that some open pit (OP) engineers have analysed underground mining methods, mainly because they are anticipating the end of the economic life of those operations in the near future. In comparing OP to UG mining methods, he said block caving was one of the lowest cost underground mining methods, which can compete with some open pits because of the high production rates, levels of mechanisation and the cost level that can be achieved. He said underground mining presents more technical risks than open pit methods with the possibility of events such as air blasts, rock bursts and hang-ups. These risks could be quantified and managed in a rational, technical and reasonable way. He described some key issues regarding block caving, some basic information requirements, cost trends, potential production capacity, management issues and the expected evolution of some techniques that could improve or solve some of the main technical constraints of the method.

Fuentes (2004) questioned why one has to analyse how many years in advance are available to consider an underground mining configuration, or why one should have to evaluate an underground mining method in an early stage of the open pit planning. Moreover, he posed a question: “Is it a crazy idea to analyse UG mining 10 years in advance of the final pit achievement?”(Fuentes (2004) in *Massmin 2004*: 633). Fuentes explained that traditionally the decision-making process in the open pit planning does not take into account the opportunity cost associated with the underground exploitation of the remaining resources left by an open pit design. Standard methodology considers sequential pushback evaluation and identifying the expansion that maximizes NPV of the design. Usually, until this time a break-even analysis (OP versus UG) was carried out using a primary approach for the underground exploitation with big uncertainties within the UG project basis. Fuentes (2004) said OP-UG transition is anticipated when the economic life of the open pit operation is nearing its end. Considering what Fuentes (2004) reported, it is however more appropriate that transition indicators are to be used during the Life Of Mine (LoM) schedules and their annual reviews so that the window of opportunity is not closed for the underground project and to derive maximum capabilities from the ore bodies.

Araneda *et al* (2004) stated that an option of a combined open pit and underground caving operation was the best long-term option to capture value. Araneda *et al* (2004) presented the overall process, the final plan and discussed some challenging issues. El Teniente is one of the largest known deposits of porphyry copper in the world and one of the five divisions of Codelco, a Chilean state-owned company. It is situated 80 km south of Santiago and 44 km up in the Andes mountains and comprises of mining, processing and smelting facilities. At El Teniente over 1,100 million tonnes of ore were mined out during almost 100 years of mining. The open pit is now in operation in the north-west side of the deposit, letting the east and south side proceed with underground mining.

He said the challenging issues regarding the open pit may be grouped into three main categories of information, interaction and planning. Interaction of personnel is certainly one of the most challenging aspects of the plan, and first among underground disciplines, but also between the open pit and underground operations. Sequencing of underground mining with the open pit mining was treated initially under a heuristic approach, however the complexity of the problem, and the fine-tuning required, forced the mine planners to treat it in a more detailed way. A four dimensional model was set up based on the 3D subsidence angles defining active caving zones and its evolution in time (fourth dimension). Although Araneda *et al*'s (2004) assertion on sublevel caving was true, the decision was based on one case study mine; however, the OP–UG transition decision should be based on several case study mines in order to have a suitable model.

2.1.5 Evaluation of technical and economic criteria involved in changing from surface to underground mining

Musendu (1995) tried to establish a general approach to determine the optimum level at which to change from surface mining to underground operations by comparing the theoretical to optimum factors that affect the transition depth. Musendu (1995) focused on the transition level at which open pit mining switches over to underground mining methods. Some of the variables he considered as affecting the sensitivity of transition depths were:

- Grade;
- Dip of the deposit;
- Size of the deposit;
- Underground recovery;
- Underground dilution;
- Underground production rate;
- Surface production rate;
- Surface fixed cost;
- Underground fixed cost;
- Surface waste cost;
- Surface ore cost;
- Underground variable cost;
- Price;
- Discount rate;
- Inflation rate;
- Plant recovery;
- Slope angle; and
- Taxation rate.

Musendu (1995) stated that transition indicators are sensitive to the following mining parameters:

- Mining recovery: favours open pit than underground due to mass production in the open pit;
- Price and grade: higher price and higher grade favour open pit mining than underground mining;
- Cost (OP and UG):the higher the OP cost the better for UG;
- Surface and underground variable ore cost: lower OP cost than UG cost favours OP;
- Cost of stripping waste: the higher this cost the earlier the transition;
- Production rate: higher production rate favours open pit mining than underground mining except when using the caving mining method; and
- Underground dilution: impacts on transition depth by reducing the grade to the plant.

Musendu (1995) considered most variables involved in the OP – UG transition but like Bakhtavar *et al* (2008) based the transition problem on transition level; hence T_L which should be T_{Lt}, where t is the point in time at which the parameters are obtained (or estimated).

The Pyhasalmi Mine is located in central Finland. The copper deposit was discovered in 1958 (Kurppa and Erkkila, 1967). The ore obtained from open pit and underground mining was being crushed by the same underground crushing facilities. The ore body extends from the surface down to at least 500 m, the length is 650 m long, the central part is 75 m wide and dips at 50°, 70° and 90° in different sections of the ore body as illustrated in Figure 2-3.

Kurppa and Erkkila (1967) in their paper on changing from open pit to underground mining at Pyhasalmi stated that a unique feature of the mine was the simultaneous mining of the open pit and underground mining as well as the long transition period. The geometry is suited for simultaneous OP-UG mining as shown in Figure 2-3. The vertical position of the ore body and the fact that it extends sufficiently deep, meant that inclined raises allowed them to direct the ore into the underground crushing plant. The heavy rubber-wheeled equipment employed in the open pit gave such good results that similar equipment was also used underground according to Kurppa and Erkkila (1967).

About 25% of the production came from underground mining. The ore body was covered by overburden of 2 to 3 m thick; hence open pit mining was the obvious choice at the commencement of activities. However, the ore body extends deep in the ground thereby allowing them to use underground mining methods for exploitation. It was calculated for the open pit mining to have ore for ten years to serve the surface operations and to allow early testing of the transition to underground mining. Waste from the open pit was used as fill for the underground operations. A sufficiently early commencement of underground stoping was to allow the dumping of waste from the lower part of the pit directly into the stopes. The establishment of an experimental stope under the open pit, the stability of which could be observed and used later for testing of the filling operation, was also important. There was enough time for the development of underground stoping. The area of the open pit was 56,000 m^2, and the pit was 330 m long and 225 m wide with average slope of the walls as 60º. Front-end loaders were used to dump the ore into the ore passes situated at the bottom of the pits at a depth of 200 m below surface. (Kurppa and Erkkila, 1967) made a valid point that the ore body configuration dictated the obvious choice of the combined mining method (both open pit and underground).

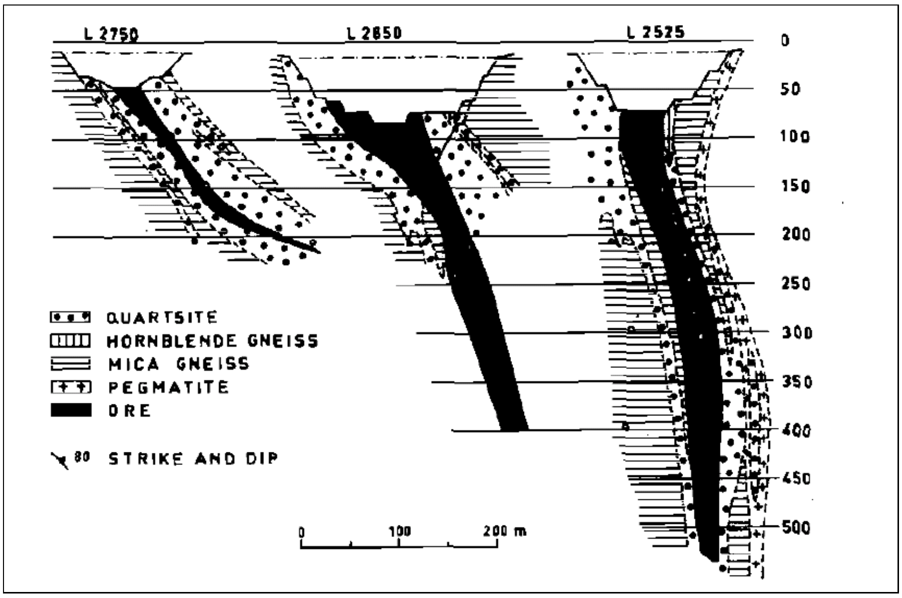

Figure 2-3: Geological sections for an ideal OP-UG transition (Kurppa and Erkkila, 1967)

Kurppa and Erkkila (1967) used the mine ore body’s configuration to prove their point, which may not be used to generalise OP-UG transition. They also stated that the transition from open pit to underground mining took place gradually over a period of several years to ensure testing of the underground operation and to ensure smooth production. The reasons for the gradual change over of the OP–UG transition at Pyhasalmi were valid but the time required for the transition could not be determined.

2.1.6 Underground mining: a challenge to established open pit operations

Arnold (1996) used the Barrick Bullfrog mine located about eight kilometres from Beatty, Nevada, as a case study to state the inherent problems in starting and operating an underground mine from an open pit mine. He stated that the ultimate pit bottom is generally within 100 to 300 metres (300 to 1,000 feet) from the surface. Arnold (1996) noted that the critical path in open pit mining flows through the drilling, to design obtained from floating cone algorithm, but with underground mining, the ore body is generally much deeper, and drilling it out to accommodate a full-scale design is impractical. He stated that time is needed to take the drill data from a geologic model to a mining plan. It has to include as much rock mechanics, ventilation, access design, and mining method work as practical. Arnold (1996) proposed that, time was required to drill out the underground resource but did not provide a solution to the inherent problems and the transition timing.

2.2 Using Whittle software to determine when to go underground

Luxford (1997) stated that the commercially available computer programmes such as Whittle 4D can now be used to determine where to make the transition from open pit to underground mining and stated the assumption that these programmes can determine the optimum final pit floor to about ± 20m accuracy. In 1998, Whittle programming (Four-X) developed an open pit and underground mining interface in the optimisation software to assist in the determination of the underground option, but the software cannot be used to determine the period of time to transition. However, management can make limited decisions based on quantified operational scenarios in the open pit to underground transition. Whittle Four-X can indicate the point at which it becomes more economically viable to proceed to underground mining, a decision which is difficult to make by traditional methods. Some of Whittle’s (2009) suggestions regarding Whittle software were as follows:

- Whittle can be used to determine the most profitable option: open pit or underground;
- Whittle can be used to indicate at which point it becomes more economical to proceed to underground mining;

- Whittle cannot be used to determine whether it is worth going underground or not;
- Whittle cannot be used to determine how much money can be gained from the underground mine; and
- It cannot be used to optimise the underground operation, schedule underground material, or include underground material in its results.

The enterprise optimisation is among the latest developments in optimisation in the mining industry. It seeks to include most value and assets in the enterprise portfolio and periods together including planning and modelling with uncertainty thereby making the impact of uncertainty measurable, managed or exploited. Whittle's (2009) theory of Enterprise Optimisation defined 10 value levers driving the decision as follows:

- Resource
- Pit optimisation;
- Pit phasing;
- Mine schedule;
- Cut-off grade and blending;
- Stockpile;
- Plant calibration;
- Product (mix and specifications);
- Logistics; and
- Market.

Whittle (2009) did not consider the underground portion and hence Whittle software remains an indicative tool for the transition to underground mining. The application of the enterprise optimisation methodology will provide answers and guidance to relevant questions to be answered. The enterprise optimisation has been introduced to reduce the uncertainty in the open pit planning but did not address the OP-UG transition timing. The question is at what point should open pit mine stop for the underground mining to commence.

2.3. OP –UG transition framework

Figure 2-4 shows the interaction between open pit and underground mining during the transition.

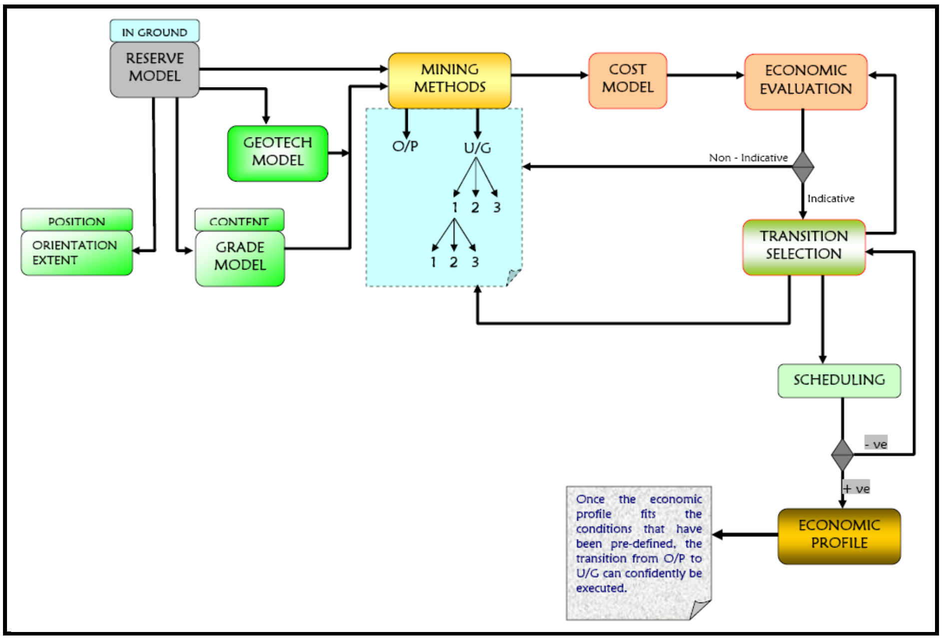

Figure 2-4: OP – UG transition framework

The OP – UG transition decision entails inter-departmental interactions as shown in Figure 2-4 to make the outcome of the transition decision acceptable to the stakeholders. Among some of the interactions involved are the grade, geotechnical properties of the rock and the mine reserves to generate the various models. UG mining method selection and scheduling are iterative and could take time to reach an optimum economic decision.

2.4. Checklist for OP-UG transition

The OP-UG transition model has the following characteristics:

- Gold as early as possible;
- Sound infrastructure with acceptable risk;
- Infrastructure must fit within a total infrastructure strategy capable of mining to desired level;
- Seamless production between the phases; and
- The highest financial returns.

To have a sound OP - UG transition model, the following checklist was developed by the researcher and requires questions to be answered during the data gathering phase before the commencement of the project. The checklist will ensure that the OP-UG transition decision is not based on an individual effort but on a team effort since the questions will involve expert input from more than three disciplines.

2.4.1 Geology

1. Is there an underground potential for the ore body?
2. Does the ore body extend beyond the current open pit surface?
3. What is the grade and depth of the ore body, which extends beyond the current open pit surface?
4. What are the thickness, dip and strike of the deposit?
5. To which resource classification has the ore body been drilled?
6. Is the ore body geometry suitable for underground mining (layout of ore body, ore zone width and geometry)?
7. Which mining methods could be used possibly for the exploitation and has dilution been taken into account?
8. What is the cut-off grade for the various mining methods?
9. What is the reserve, and how fast will it be mined?
10. What is the value generation potential for the total mine from UG by analysing OP mining only, combination of OP and UG mining and UG mining only?
11. What is the incremental value of various stripping ratios in the OP vs. UG – should the pit finish early?
12. Should the UG mine selectively or do bulk mining to full ore body width (as this depends on the type of ore body)?
13. Where does the high grade ore sit in relation to the ore body, can the mine be high graded, what is the open pit reconciliation like?

2.4.2 Operational

1. Will the existing resources for the open pit (equipment and personnel) be utilised for the underground mining?
2. Does the mine have enough time and spare resources to build a OP – UG transition stockpile to see the mine through the teething period of the transition?
3. Can the equipment available for the open pit mining be converted for the underground?
4. Can the mine achieve a reasonably high and consistent profile for the life of mine (LOM)?

5. What is the LOM of the mine reserves for the current operation?
6. Is the capital cost required to transition low enough and affordable?
7. Does the country have the resources to sustain supplies such as fuel, power and water?
8. Does the mine have available expertise within the region for the transition?
9. What transition constraints does the mine have? – Minimum LoM of 5 years, does the country have laws that do not allow either type of mining, what is the minimum cut-off grade required?
10. Has the mine factored in the time it takes for an underground mine to get off the ground from concept to actually mining as this can take several years?
11. Has the mine done the geometallurgical testing to assess the different mineralogy between open pit and UG of ores containing predominantly oxides vs. predominantly sulphides? (This may affect processing).
12. Does the mine have the location of shaft or decline starting positions?

2.4.3 Geotechnical

1. In terms of geotechnical considerations, one needs to know the guidelines in terms of mining width, span, hydraulic radius, support requirements. This can drive the mining method and ultimately the costs. Are there any regional instability issues posed by underground mining?
2. Do the ground conditions allow for the transition, bearing in mind the potential of large structures that can cut off the access to the ore body?
3. Does the mine have a geotechnical database such as Rock Mass Rating with enough confidence and how was it derived?
4. Is there a good database of Uniaxial Compressive Strength (UCS) available?
5. Will the available geotechnical data or information be enough to be used to derive or calculate the stable spans in the underground mining and pillar strength?
6. Are the structural trends generally consistent along both the strike and with depth?

7. Will the groundwater inflows be relatively low and manageable at reasonable cost, bearing in mind the water that was ponded in the mined out pit?
8. Are there known major adverse faults?
9. Are the regional stresses and principal mining induced stresses amenable by considering the weight of the unmined material on the crown pillar?
10. Will the geothermal gradient (rock temperatures) expected to be unduly high?

2.5 Chapter summary

This chapter described the previous work done by various authors to address the OP-UG transition problem. The commercially available computer programmes such as Whittle FourX® can now be used as an indicative tool to determine the transition from open pit to underground mining, but given the assumptions involved in these programmes they will probably determine the optimum final pit floor to ± 20m. Stacey and Terbrugge (2000) suggested transition timing of up to 20 years for the planning and implementation period for transition from OP-UG. These authors provided vital points on open pit to underground transition but did not provide transition criteria to guide the OP-UG transition. OP-UG transition model checklists were developed in the form of questions to be answered during the data collection phase before the commencement of the transition project. To address the uncertainties in the geological models, simulated models will be used in subsequent chapters for the OP – UG transition model. The next chapter will explain the modelling process adopted for the OP-UG transition.

3.0 PROCESS FOR MODELLING OPEN PIT TO UNDERGROUND TRANSITION

To maximise the return on mining investments, the options regarding the choice of mining method (whether to go open pit or underground) should be analysed early in the mine life as well as during the annual reviews in the life of mine (LoM) schedules. The basic problem that most mining companies have faced is the lack of a model or methodology to follow that will address these challenges. This research was undertaken to develop a model for mining companies to address the OP-UG transition challenge. The model uses simulated models to address the geological risks due to grade uncertainties.

Four case study mines with underground potential were selected. These case study mines were limited due to the confidentiality associated with use of geological block models, which are used by the companies to declare their annual reserves. There were also time constraints needed to create and run these models. The four case studies were used to develop the OP-UG transition model. The case studies were all selected from gold deposits. The geological models for the four case studies deposits were handed over by the Geology section of AngloGold Ashanti corporate office in Johannesburg while the mines involved assisted with the site information on cost and optimisation parameters needed for the project. The details of the data in each geological block model are summarised in Appendix 1. In order to quantify risks in the grade estimates, conditional simulation models were generated using Direct Block Simulation methodology (DBSim) for the case study mines. One of the geological block model for the four case study mines was already a simulated model as received from the mine.

3.1 Processes followed in the creation of OP-UG transition model

Geological block models received from the mines were used to create the simulated models. The geostatistical parameters used to create the simulated models were site specific; hence, care was taken to validate the models against other estimation techniques. The guidelines followed in the process of generating the simulated models were prepared and checked by the AngloGold Ashanti geological teams. The processes used for the OP-UG transition modelling were as follows:

- Models and drill holes validations using Datamine® software;

- Creation of simulation models of the AU grade variable using Direct Block Simulation methodology (100 simulations for 4 case study mines);
- Preparation of the simulated models for pit optimisation;
- Pit optimisation using Whittle® software (100 shells for 4 case study mines);
- Scheduling using XPAC® software;
- Generating results of OP-UG transition indicators; and
- Analysis and interpretation of the results.

3.1.1 Model preparation for simulation

The geological block models and the drill holes used were prepared using Datamine software, one of the General Mining Packages (GMP). The model handover notes defining various attributes in the block models were provided by the geological teams. The geological block models were checked for errors before the block simulations. The checks were done on the block models and the drill hole data to ensure that there were no missing or pre-determined values in the density field. Visual checks were also done to identify the missing blocks in the block models and the drill hole samples. Datamine macros in Appendix 2 were written by the researcher and used to check and validate the models and the drill hole samples. The Datamine macros in Appendix 2 were used to evaluate the geological models before the simulation. Some of the geological block models received from the various case study mines were recoverable resource models. However, for the purpose of the research, the panel grade values (AU) in the models were accepted to represent the grade values because the two were close enough as shown in the grade tonnage curves in Figure 3-1. Again the recoverable uniform condition models (UC) and the panel grade (AU) values were close enough and well reconciled. Geita Nyankanga case study showed an increase of 18% in tonnage and a decrease of 9% in grade in the uniform condition (UC) proportion fields. Grade tonnage curves were used to analyse the characteristics of the various ore bodies before and after the block simulation to confirm the simulations methodology. Figure 3-1 and Figure 3-2 show the grade tonnage curve for Morila ore body before and after simulation, respectively, while Figure 3-3 and Figure 3-4 show the equivalent for Cerro Vangurdia Mine (CVSA). Only five out of the hundred simulated model results were plotted on the grade tonnage curves for Morila and CVSA for eligibility. The data for the grade tonnage curve are shown in Appendix 3.

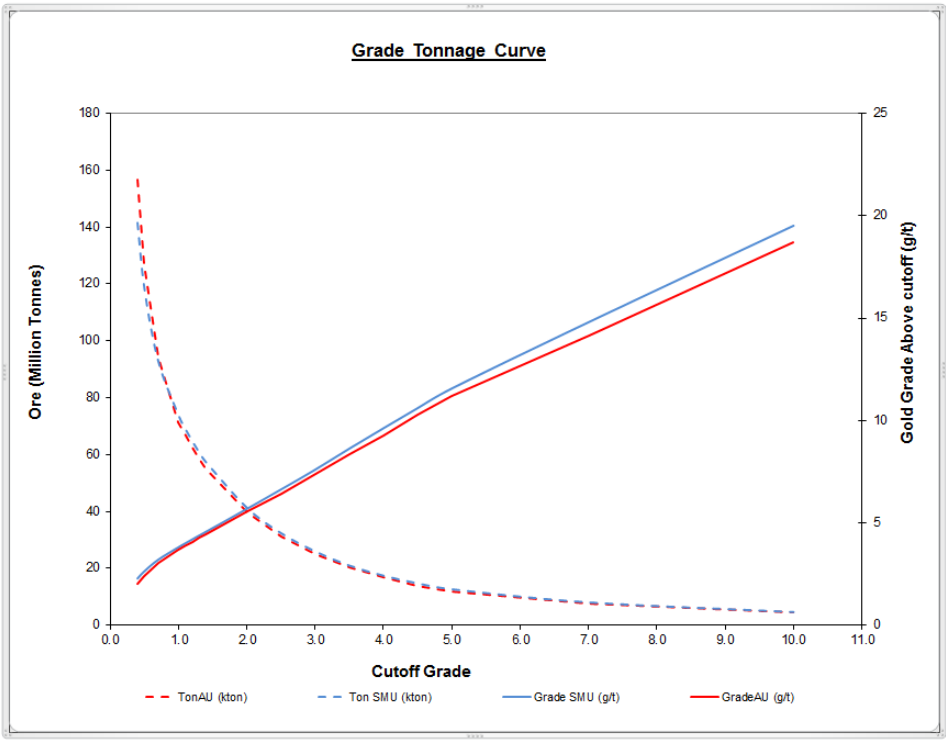

Figure 3-1: Grade tonnage curve of Morila ore body with UC and kriged models

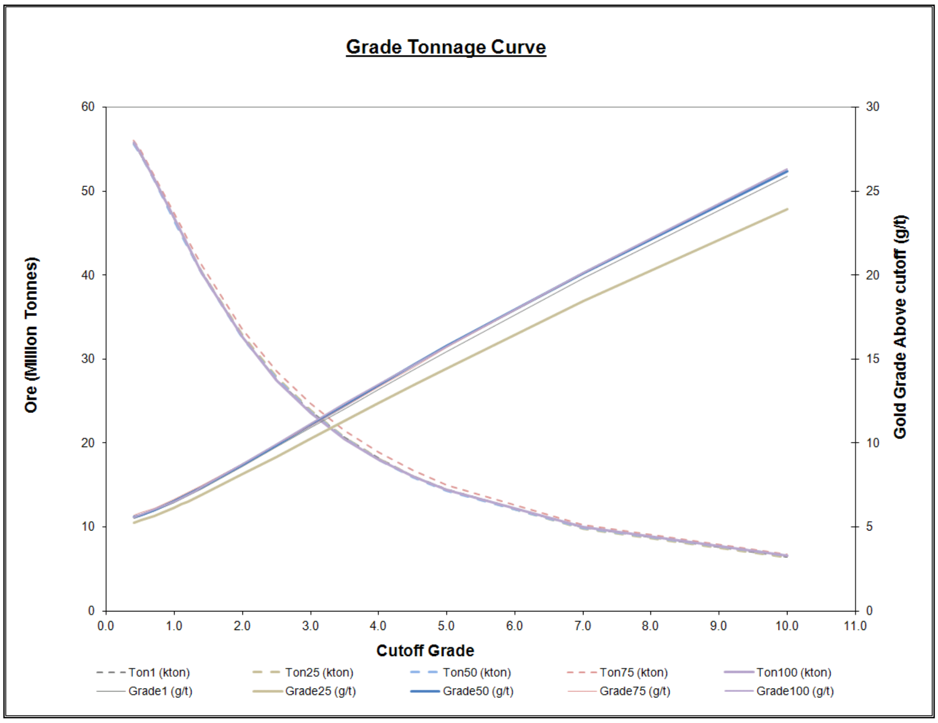

Figure 3-2: Grade tonnage curve of Morila ore body showing the simulated AU values

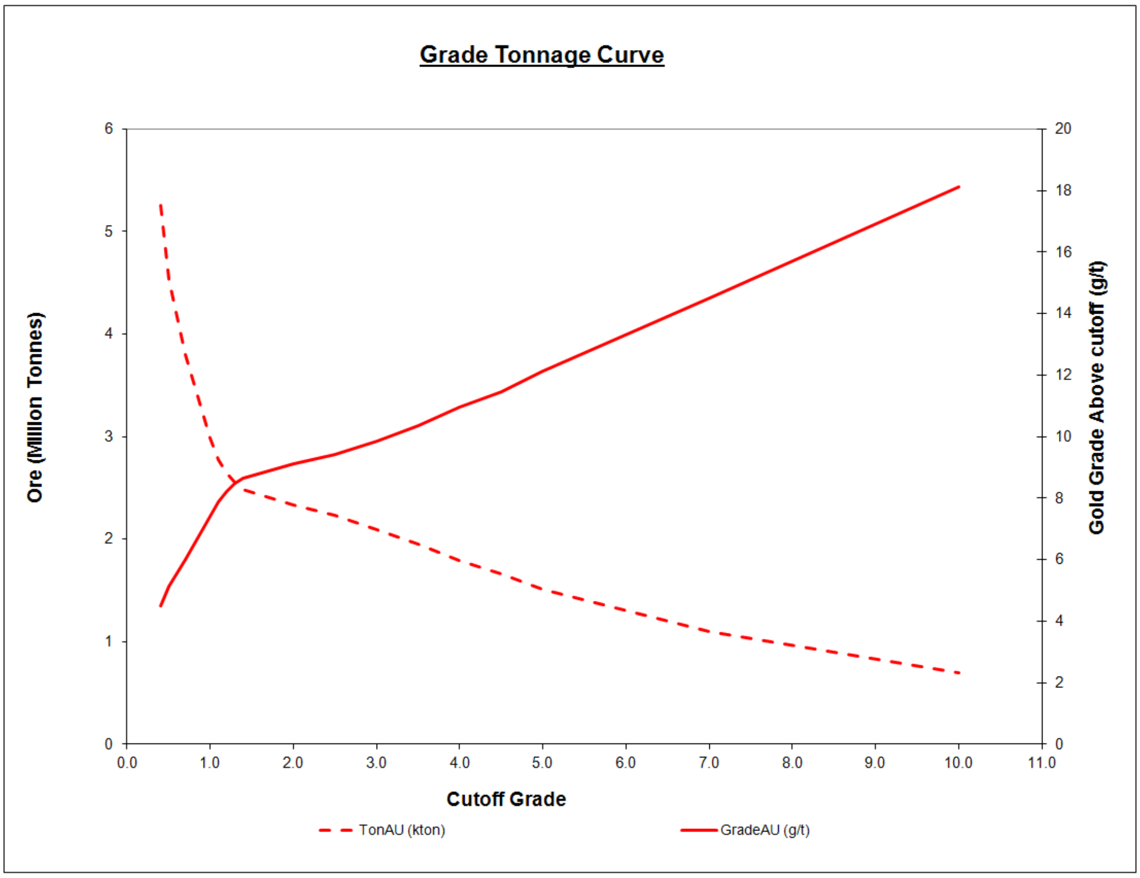

Figure 3-3: Grade tonnage curve of CVSA ore body with kriged model

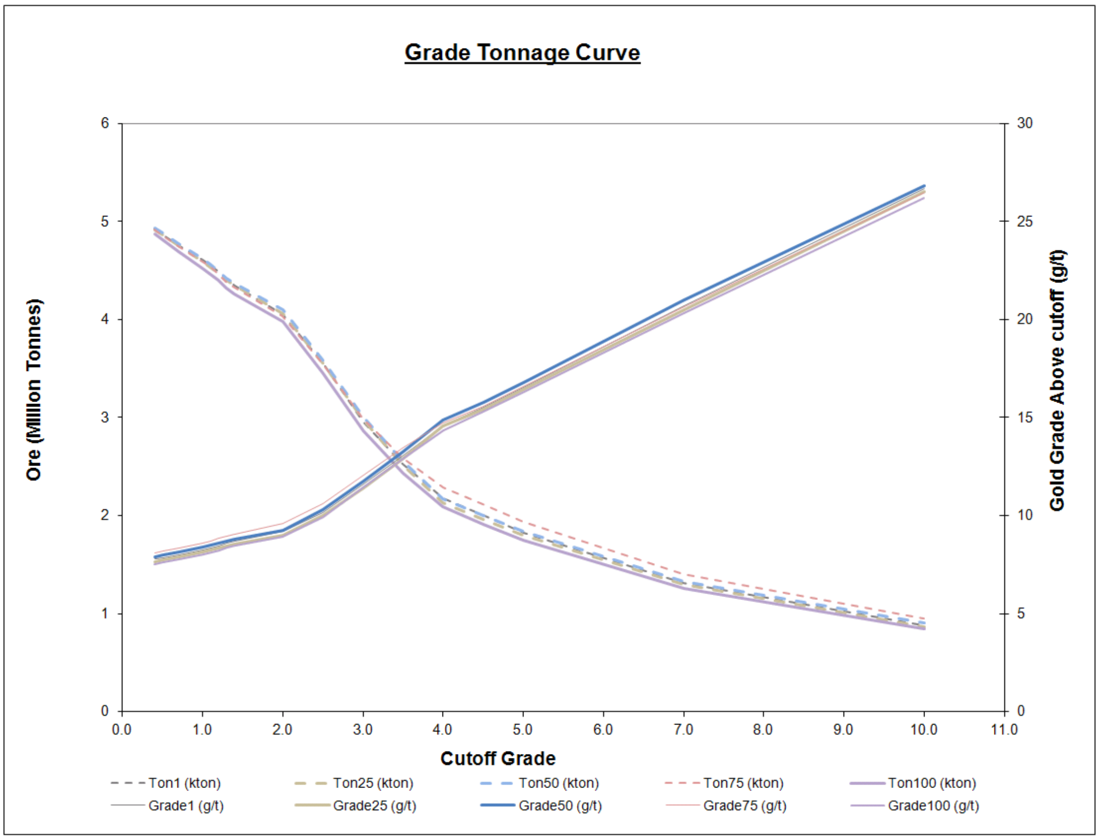

Figure 3-4: Grade tonnage curve of CVSA ore body showing the simulated AU values

3.2 Direct block simulation

Direct-Block Conditional Simulation (DBSim) is one of a number of geostatistical simulation techniques that produce a range of equi-probable realisations of the likely configuration of the mineralisation in an ore body. Benndorf and Dimitrakopoulos (2009) demonstrated the application of conditional simulation techniques for modelling ore bodies by the use of efficient algorithms, due to the large number of grid nodes in order of tens of millions of blocks. Peattie and Dimitrakopoulos (2009) stated that the direct block simulation was an efficient and practical method. According to Dimitrakopoulos and Luo (2004), conditional simulation was seen as an extension of the group sequential Gaussian simulation.

A Gaussian variogram model is not necessarily used in the Uniform Conditioning process, as the change of support and grade estimates are based on a variogram calculated on raw, or non-transformed data. The change of support model for both processes uses the Discrete Gaussian model, which is based on the assumption that it is not fully true but constitute an approximation to the exact solution based on Krige's relationship in Equation 3 -1.

$$\sigma^2(v/D) = \sigma^2(o/D) - \bar{\gamma}(v,v)$$ **Equation 3-1**

Where $\sigma^2(v/D)$ is the dispersion variance of blocks within the deposit.

$\sigma^2(o/D)$ is the variance of the grades of samples of all possible positions o and D in the deposit.

$\bar{\gamma}(v,v)$ is the value of the variogram in small volume v, within the block.

D is the deposit.

v is the dispersion unit.

Uniform conditioning relies on a single, conditionally unbiased estimate of the block grade, and can be estimated using Ordinary Kriging. Simulation techniques often work more effectively using Simple Kriging in the Gaussian-space in which the simulations are performed; more weight is applied to the mean (zero) when the local conditioning data is widely spread. One of the major sources of risk to mining not achieving its production target is uncertainty in the expected ore grade and tonnage (Dimitrakopoulos, Farrelly and Godoy, 2002). In order to handle the geological variability, simulated ore body models were used to determine the impact of transition timing. Figure 3-5 shows workflow comparison between uniform condition and direct block simulation.

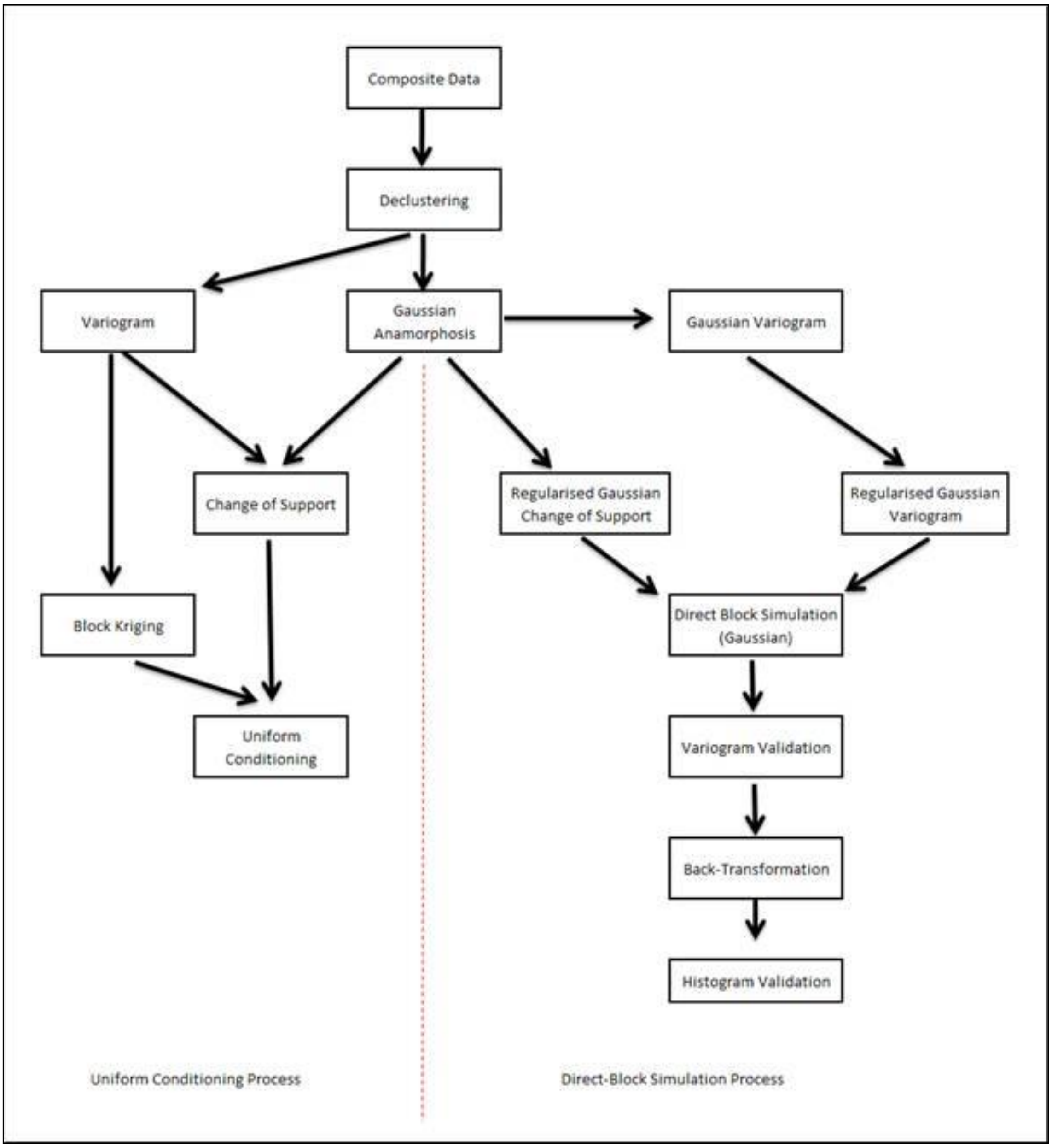

Figure 3-5: Workflow process for uniform conditioning and direct block simulation

The capabilities of Isatis software were fully utilised to achieve the recommended steps used in the creation of the simulated models for the study. The drill hole, wireframe and the Kriged models for Morila, Sadiola and Geita mines' deposits as received from AngloGold Ashanti were used to generate the simulated models utilising the capabilities of direct block simulation module in the Isatis software.

3.2.1 Quality control and simulation data coverage

Simulation quality control was achieved by the use of histograms and experimental variograms for all the simulated models for each case study deposit. Conditional simulation tends to be data driven when the conditioning data are sparse, there is less data charging of the simulations, and models are much more sensitive to the way variogram parameters are used by each algorithm. As a result, a decision was taken not to deplete the already mined-out information from the geological models prior to the simulation. Simulated models were generated for Geita, Sadiola and Morila case studies whilst the CVSA simulated models were done by the site geologist. Various authors have used different numbers of realisations to model uncertainty. Dimitrakopoulos, Farelly and Godoy (2002) considered that 50 realisations were sufficient for their purposes while Goovaerts (1997) used 100 realisations. The 100 realisations used for this study produced more stable results than fewer realisations.

3.3 Methodology for creating simulated models

There are many methods available in the creation of simulated models for geological risk quantification. The researcher used the Isatis version 11.01 software, which uses the Direct-Block simulation method (DBSim) to produce the simulated models since a licence was available to the researcher to use the software. The Direct-Block Conditional Simulation uses a Gaussian variogram model, which is regularised to the smallest mining unit (SMU) support. The regularization largely nullifies the nugget effect as seen in the variograms calculated on composited data.

A variogram is the structural tool that helps to generate the simulated model. It is necessary to fit a variogram model to the experimental point variogram. Variograms of the simulated Gaussian values can be calculated to compare to the input block Gaussian variogram. All three directions can be calculated at the same time, however, when run from a journal file, one can only save (or print) a single direction as a graphic file, therefore it is necessary to run in a loop for each direction. The input file is the macro variable of the Gaussian simulated variables. The variograms are compared to the input block Gaussian variogram model and stored as an output experimental variogram file. Figure 3-6 shows Gaussian point variogram window, Figure 3-7 shows the point variogram fitting window while Figure 3-8 shows the variogram validation window. Summary steps followed in creating the simulated models for the case study mines are summarised in Appendix 4. Figure 3-9 shows the workflow diagram for creation of the simulated models with the direct block simulation method using the Isatis software.

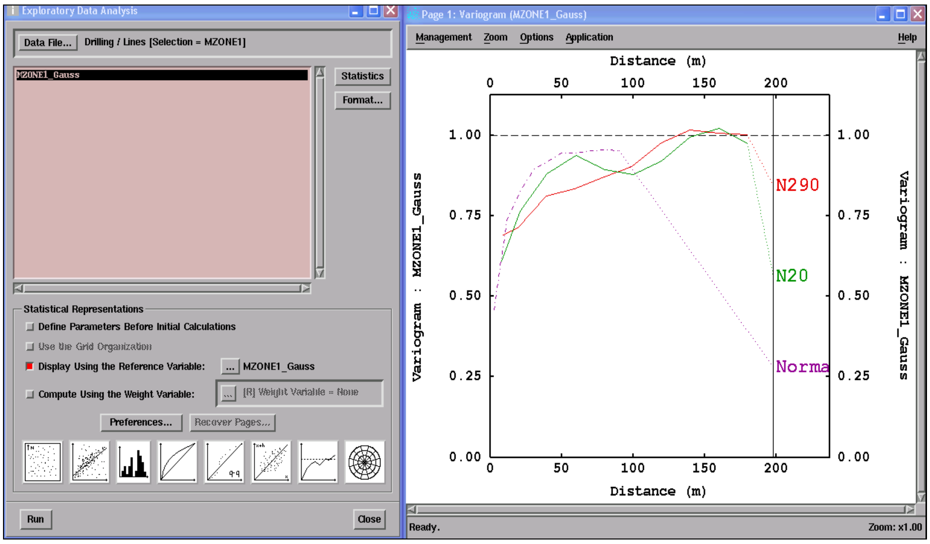

Figure 3-6: Gaussian point variogram window

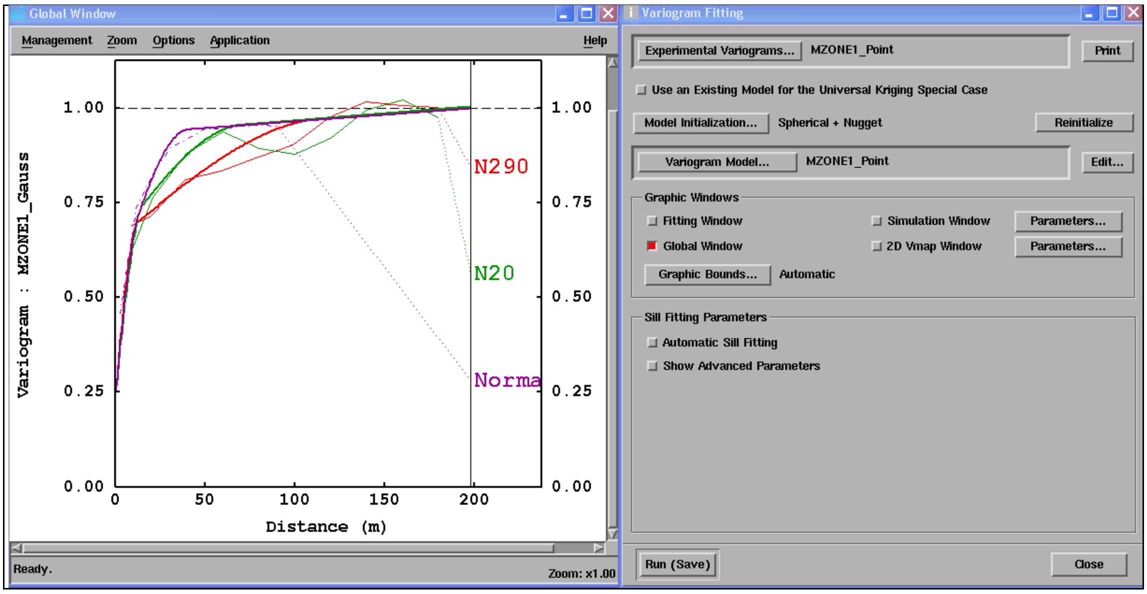

Figure 3-7: Point variogram fitting window

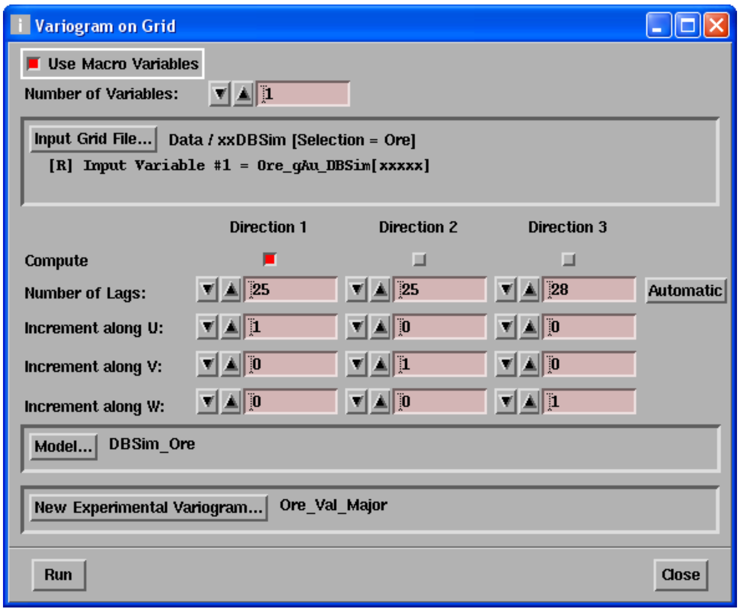

Figure 3-8 : Variogram validation window

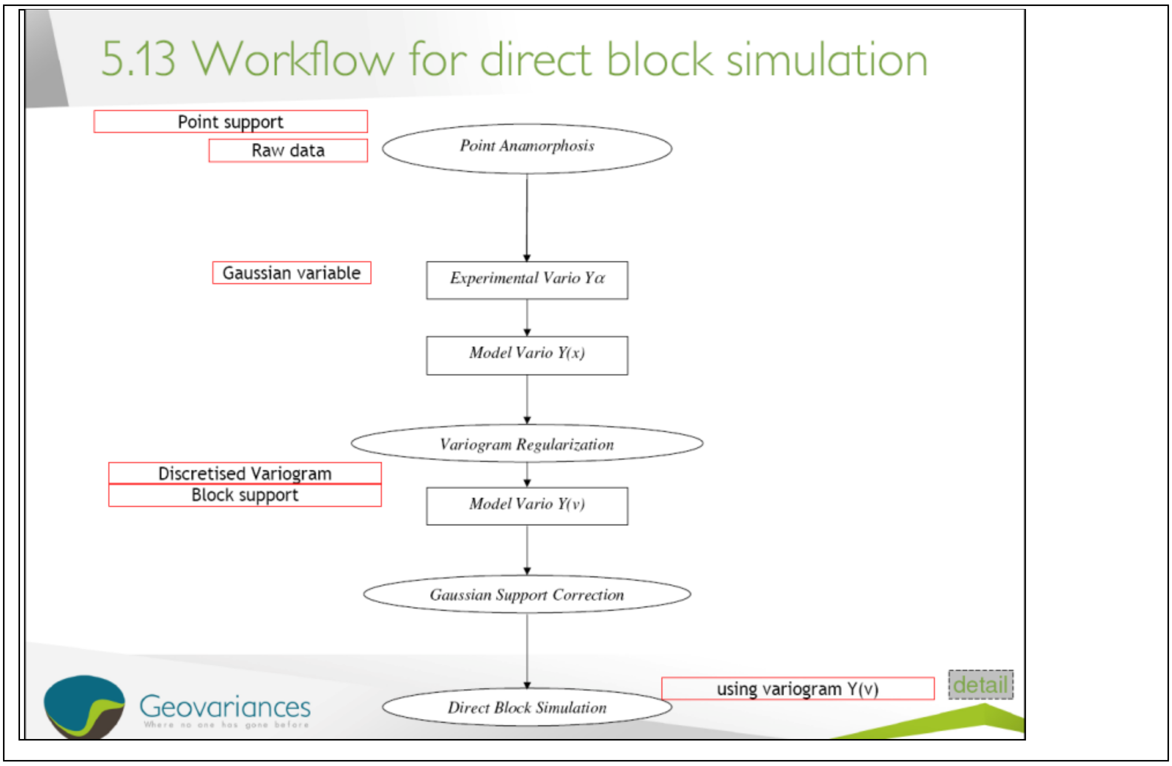

Figure 3-9: Workflow for creation of simulated models with the direct block simulation method (Source: Geovariances)

3.3.1 Problems encountered in simulated models creation

Most mining companies are aware of the benefits and confidence that simulated models will add to their mines' resource models when used to quantify risk in mine planning. However, many mining companies lack the support and methodology fot its implementaton. The mining industry needs to have a systematic approach with stepwise methodologies to follow to remove the bottlenecks. Some key advice to help gold companies intending to implement the use of simulated models is as follows:

- Geological models created to use for simulated models creation must be Kriged with recommended parameters suitable for simulated models;
- Geological block models for simulations must be checked and well validated;
- Models to be simulated must have a field defining the mineralisation envelopes;
- Simulated models of the AU grade can be created using different simulation programs and software;
- The use of models with sub-cells must be regularised before simulation to reduce simulation running time;
- Simulations are to be done for separate kriged zones (KZONES) for both ore and waste and in batches;
- The simulated results must be exported after the simulation with the density and grade to the required General Mining Packages (GMP); and
- Simulation results are different for each deposit and depends on parameters used and the simulator.

3.4 Preparation of simulated models for pit optimisation

The simulated models were prepared for pit optimisation using the appropriate macros. The Whittle input files for the four case study deposits (100 simulated models per deposit) were all generated using the macros. The models were imported individually into Whittle with summaries of the rock tonnes and ore tonnes from Whittle compared to the results from Datamine. All assumptions were site specific. Figure 3-10 shows the Datamine macros used to prepare the simulated models before optimisations.

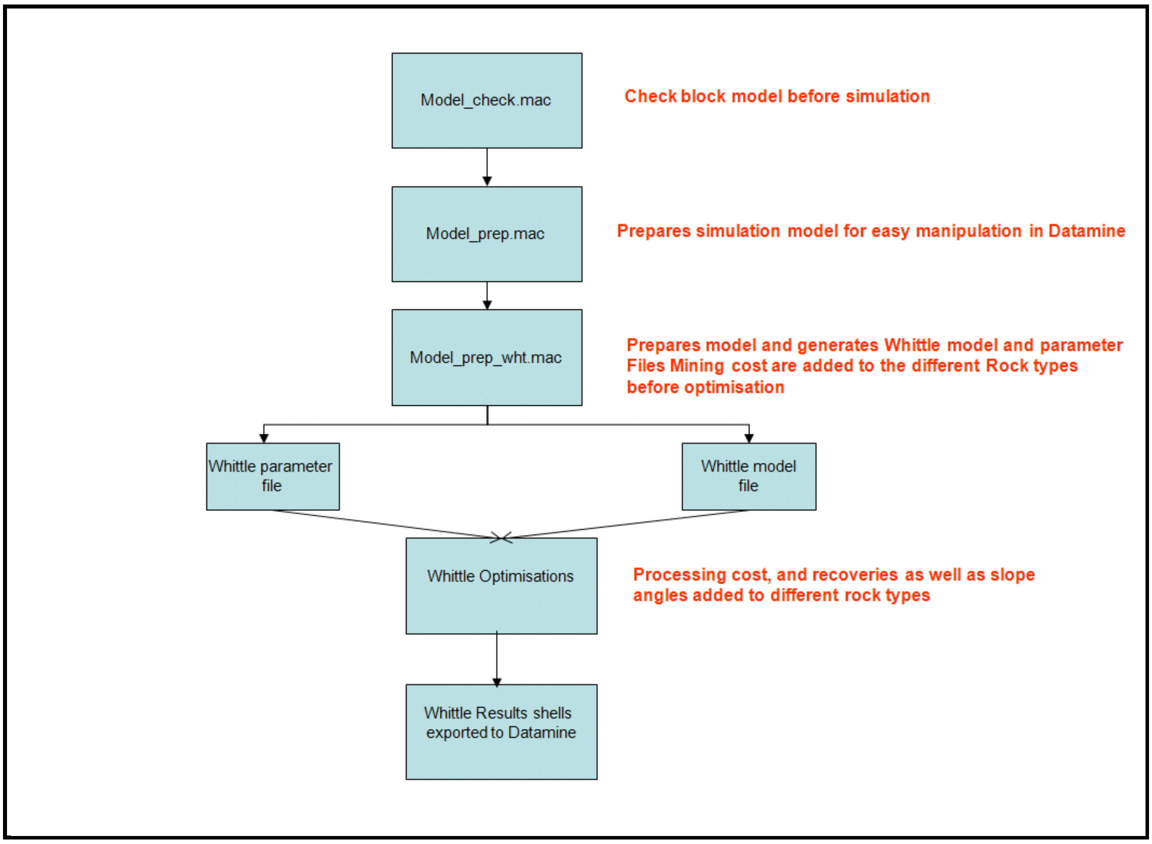

Figure 3-10 : Flowchart of model preparation macros

3.5 Optimisation of open pit and underground mining

An optimal strategic plan for an open pit mine maximizes NPV while meeting a wide range of production, engineering and economic constraints. This is done to identify the correct limits of the open pit when the underground mining is considered as an option for all the case study deposits to have the highest value. The objective of the pit optimisation is to maximise the cumulative value of ore that could be mined and processed. The OP-UG transition model starts with pit optimisation to determine the optimum limits and size of the pits to be mined. Optimisation in this study was done using Whittle software which uses the Lerch-Grossman algorithm to progressively construct a list of blocks that should be mined using a set of assumptions (mining cost, processing cost and recoveries, metal prices, and slope angles). The final pit outline includes blocks that are worth mining and excludes uneconomical blocks. Whittle creates a series of nested shells by varying the revenue factors. A pit shell is selected for design using a graph plotted with the cash flow and the revenue with the ore tonnes. The ore tonnes and waste tonnes from the optimisation results are substituted into Equation 3-2 to calculate the cost per ore tonne. To decide whether the pit is best suited to be mined as an open pit up to the optimum point during the optimisation depends on the assumptions used for the optimisation. The optimisations in this study were done using a long-term gold price of USD 1300/oz and then compared to pits designed by the mine at a gold price of USD 850/oz. The required pit shell is selected to correspond to the equivalent cost per ore tonne shell or based on the revenue factor and the stripping ratio as shown in Table 3-1.

$$\boldsymbol{Cost\ per\ ore\ tonne}\ (\$) = \frac{\boldsymbol{Cost\ of\ mining}(\$)\times(\boldsymbol{ore\ tonnes} + \boldsymbol{Waste\ tonnes})(\boldsymbol{t})}{\boldsymbol{Ore\ tonnes}(\boldsymbol{t})} \qquad \textbf{Equation 3-2}$$

The underground mining cost per ore tonne was estimated at USD 54 per tonne based on site information of actual cost. The Whittle optimisation exercise calculates the optimal pit at the reserve gold price and also calculates larger pits based on an increasing gold price. By using the larger pits at increased revenues the open pit size and mining cost at USD 54 per ore tonne can be determined. Table 3-1 shows the calculation of how the underground option was selected based on the cost of mining the ore per tonne from underground.

The underground option for this study was created using the Whittle interface portion. The selection is done by identifying the rock types that might be mined from underground, making an estimate of the underground mining and processing costs together before assigning it to the processing cost with the processing recovery. Each of the 100 Whittle models from the 100 simulated grade fields were imported separately. Optimisations were done to create 100 pit shells (one per realisation) for all the case study deposits. After running the models for several times a decision was made to reblock the model in Whittle to reduce the processing time per model. Reblocking was done in the X and Y direction to increase the block size from 10 x 10m to 50 x 50m. The shells for each run were exported to Datamine software (as .res, .par and .dxf) files. For this thesis, pit selections were not done but shells with revenue factor of 1 were selected for each simulated model grade value. The mining costs applied in the optimisation included all costs associated with mining such as, loading and hauling; drilling and blasting; pit dewatering; grade control drilling; and mining overheads. The quality and reliability of the geotechnical data is important for the stability of the open pit walls and for the stability of the openings underground. The stability of the surface crown pillar defines the bottom of the open pit. There is a minimum thickness of the surface crown pillar required to start an underground mining operation. The underground processing stream for the relevant rock types ticked in Whittle is as shown in Figure 3-11.

Table 3-1: OP-UG transition interface calculations

Revenue factor	Pit No	Gold price (USD/oz)	Mining cost ($)	Ore tonnes (t)	Waste tonnes (t)	Stripping ratio	Cost per ore tonne ($)	Option
0.346	1	450	-2.4	71163	601353	8.45	-22.68086506	
0.384613	2	500	-2.4	182581	1856528	10.17	-26.80378353	
0.423226	3	550	-2.4	924431	15559144	16.83	-42.79451901	
0.461839	4	600	-2.4	1044941	17059957	16.33	-41.58297473	
0.500452	5	651	-2.4	15432526	294113380	19.06	-48.13924658	
0.539065	6	701	-2.4	24614832	482307823	19.59	-49.42606848	
0.577678	7	750	-2.4	30738338	593380163	19.3	-48.7301689	
0.616291	8	800	-2.4	35080180	682249314	19.45	-49.07588232	
0.654904	9	850	-2.4	41140081	864147839	21.01	-52.81202553	
0.693517	10	900	-2.4	42602309	910585303	21.37	-53.69780001	
0.73213	11	950	-2.4	45366663	982060366	21.65	-54.35323444	UG Option
0.770743	12	1000	-2.4	46907860	1036727065	22.1	-55.44324171	
0.809356	13	1050	-2.4	47290780	1049017216	22.18	-55.63746655	
0.847969	14	1100	-2.4	47673461	1059057947	22.21	-55.71559781	
0.886582	15	1150	-2.4	50475800	1151774718	22.82	-57.16405175	
0.925195	16	1200	-2.4	52120425	1225434565	23.51	-58.82783911	
0.963808	17	1250	-2.4	52388363	1233129676	23.54	-58.8917675	
1.002421	18	1300	-2.4	56312178	1400299864	24.87	-62.08015788	OP Option
1.041034	19	1350	-2.4	56857734	1422591150	25.02	-62.44844935	
1.079647	20	1400	-2.4	57264799	1449097354	25.31	-63.1324868	
1.11826	21	1450	-2.4	57387895	1453725753	25.33	-63.19577944	
1.156873	22	1500	-2.4	60153798	1606494857	26.71	-66.49549829	
1.195486	23	1550	-2.4	60355266	1618407355	26.81	-66.75524039	
1.234099	24	1600	-2.4	60454445	1620701985	26.81	-66.74075714	
1.272712	25	1650	-2.4	61813727	1710423446	27.67	-68.80946064	
1.311325	26	1700	-2.4	61912629	1715086897	27.7	-68.88415064	
1.349938	27	1750	-2.4	61959643	1716080881	27.7	-68.87220537	
1.388551	28	1800	-2.4	62170723	1723916099	27.73	-68.94898701	

Whittle requires the following parameters during its computations. These are revenue (R), selling value (SV), block grade (BG) and block value (BV). Block Value is calculated for OP and UG by multiplying the block tonnage with mining cost and mining cost adjustment factor. If the block value is positive the material will be left for UG to mine, otherwise the OP will mine it. Parcel value is calculated for open pit and UG by multiplying the processing cost by block tonnage, mining dilution and mining recovery. UG mining does not consider mining dilution or mining recovery during its processing. The revenue (R) is calculated by multiplying the available metal, mining recovery, gold price and processing recovery. Selling Value (SV) is calculated by multiplying the available metal by mining recovery, the selling cost and the processing recovery whilst the BV is calculated by adding the parcel value to the revenue minus the selling value. Figure 3-12 shows the summary of the Datamine macros used to prepare the simulated models after pit optimisation.

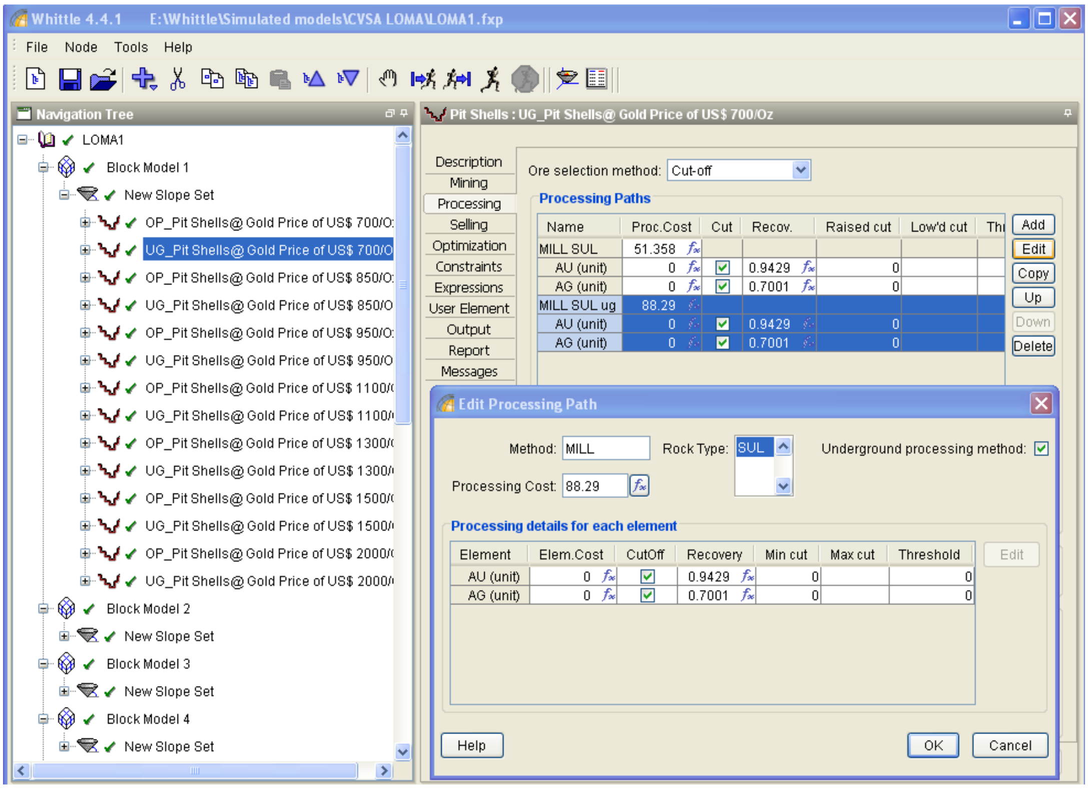

Figure 3-11: Whittle underground processing option window

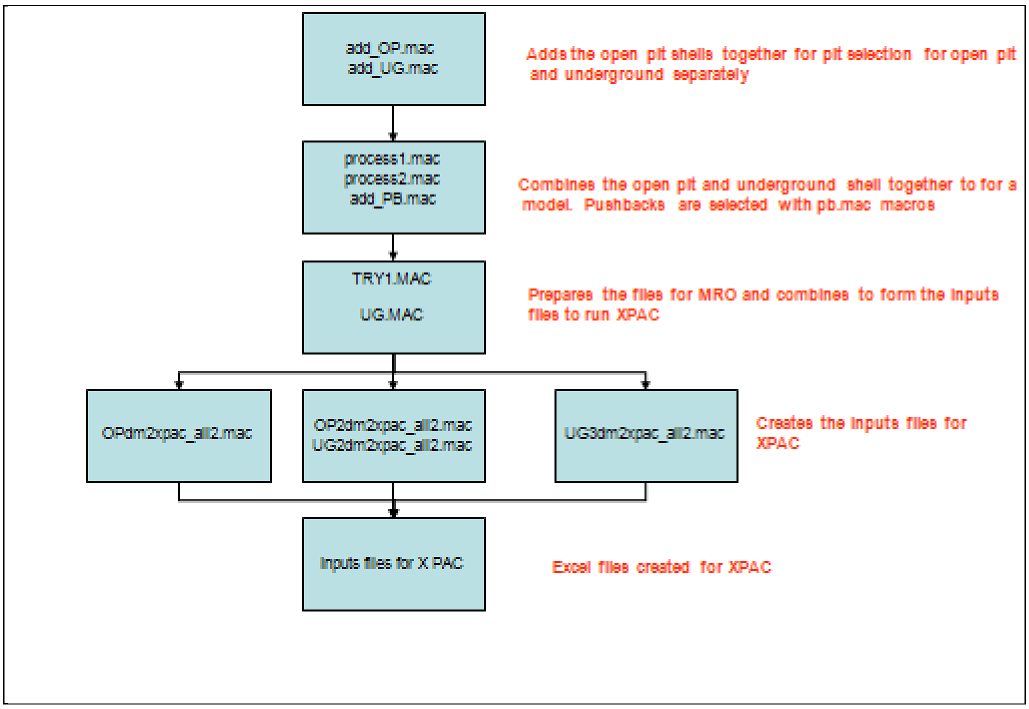

Figure 3-12: Flow diagram of macros used to prepare models after optimisation

3.6 Mineable Reserve Optimiser processes

To create a mineable stope for the underground. Datamine's MRO script was used to generate a diluted mineable model for the underground stopes. This process assesses whether resource model blocks meet a series of criteria including minimum stope width, cut-off grade, and head grade. The raw resource model as supplied from mines contained known mineralisation grades and block sizes that best modelled the mineralization. However, this model in its raw form was considered unsuitable for underground mining purposes, as it was unlikely that the largest block size in the model would be able to be mined without considerable dilution. Dilution was applied using Datamine's MRO, which agglomerates groups of blocks into larger blocks, determining the new block grades based upon the original block grades plus the included waste material (both internal and external waste). Appendix 5 shows MRO Datamine script input parameters windows.

UG mining method selection plays important part in OP-UG transition. A suitable underground mining method capable of displacing an open pit mining to fill the gold gap created for the plant feed is always a concern during OP-UG transition. The criteria used to select the underground mining methods for the OP-UG transition were the properties of the deposit: ore strength; rock strength; shape; size; depth and dip. The primary geological factors vital in selecting mining methods for given deposits are the dip (inclination to the horizontal) and thickness of the deposit. By inclination, a deposit may be classified as being flat, inclined or steep. Factors affecting selection of mining method are the size of ore body, continuity of ore body, attitude of ore body, depth of ore body and rock hardness. A 40 m crown pillar assumed between the open pit and underground mining based on geotechnical assessments for all the case study mines.

3.7 Scheduling using XPAC software

Scheduling is a critically important part of mining ventures as it deals with the efficient management of cash flows. It allocates available resources to activities over time based on the company's strategic objectives. Scheduling was done to determine the practicalities of the mining sequence, plant feed and required equipment as well as their replacement times. Scheduling was carried out using XPAC® scheduler software. The choice of XPAC as the scheduling software for the OP-UG transition was based on license availability and the researcher's experience in using it. The scheduling strategy for the OP-UG transition considered the ore treatment rate, maximum vertical mining rate, commencement of mining block and development of the main starter pits and stockpiling of ore over life of mine. Appendix 6 shows some of the results from XPAC. XPAC has four main components to create the schedule. These are:

- Main database;
- Calendar;
- Scenarios generation; and
- Results.

The Datamine macros in Appendix 7 were used to prepare the models for scheduling. Table 3-2 and Table 3-3 show samples of the mining calendars for Geita used for scheduling the mining options. Option 1 shows mining the ore body from open pit only, Option 2 represents mining the ore body from both open pit and underground and Option 3 shows mining the entire deposit from underground.

Table 3-2: Mining calendar for Geita Option 1

Year	CUT1	CUT2	CUT3	CUT4	CUT5	CUT6	Total Tonnes	No of Excavators
YEAR1	16,784,057	133,627					16,917,684	2
YEAR2		33,835,369					33,835,369	4
YEAR3		33,835,369					33,835,369	4
YEAR4		16,917,684	16,917,684				33,835,369	4
YEAR5		16,917,684	25,376,526				42,294,211	5
YEAR6		4,192,509	21,184,018	16,917,684			42,294,211	5
YEAR7			25,376,526	16,917,684			42,294,211	5
YEAR8			16,092,506	26,201,705			42,294,211	5
YEAR9				42,294,211			42,294,211	5
YEAR10				42,294,211			42,294,211	5
YEAR11				42,294,211			42,294,211	5
YEAR12				42,294,211			42,294,211	5
YEAR13				33,835,369	8,458,842		42,294,211	5
YEAR14				16,917,684	25,376,526		42,294,211	5
YEAR15				16,917,684	25,376,526		42,294,211	5
YEAR16				10,902,963	31,391,248		42,294,211	5
YEAR17					25,376,526	16,917,684	42,294,211	5
YEAR18					6,895,514	35,398,697	42,294,211	5
YEAR19						42,294,211	42,294,211	5
YEAR20						42,294,211	42,294,211	5
YEAR21						42,294,211	42,294,211	5
YEAR22						33,835,369	33,835,369	4
YEAR23						33,835,369	33,835,369	4
YEAR24						26,424,389	26,424,389	3
	16,784,057	105,832,242	104,947,261	307,787,617	122,875,183	273,294,139	**931,520,499**	

Table 3-3: Mining calendar for Geita Option 2

Year	CUT1	CUT2	CUT3	CUT4	CUT5	CUT6	UG	Total Tonnes	No of Excavators
YEAR1	16,538,398	379,286						16,917,684	2
YEAR2		33,835,369						33,835,369	4
YEAR3		33,835,369						33,835,369	4
YEAR4		16,917,684	16,917,684					33,835,369	4
YEAR5		16,917,684	25,376,526					42,294,211	5
YEAR6		2,468,188	22,908,338	16,917,684				42,294,211	5
YEAR7			25,376,526	16,917,684				42,294,211	5
YEAR8			12,621,071	29,673,140				42,294,211	5
YEAR9				42,294,211				42,294,211	5
YEAR10				42,294,211				42,294,211	5
YEAR11				42,294,211				42,294,211	5
YEAR12				42,294,211				42,294,211	5
YEAR13				33,835,369	8,458,842			42,294,211	5
YEAR14				16,917,684	25,376,526			42,294,211	5
YEAR15				5,330,921	28,504,448	8,458,842		42,294,211	5
YEAR16					25,376,526	16,917,684		42,294,211	5
YEAR17					21,629,617	20,664,594		42,294,211	5
YEAR18						42,294,211		42,294,211	5
YEAR19						42,294,211		42,294,211	5
YEAR20						25,376,526		25,376,526	3
YEAR21						16,917,684		16,917,684	2
YEAR22						5,568,850	3,000,000	8,568,850	1
YEAR23							1,032,519	1,032,519	
	16,538,398	104,353,581	103,200,146	288,769,325	109,345,960	178,492,603	4,032,519	**804,732,531**	

3.8 The validity of software used for OP-UG transition

Mining software was an important part in the success of the OP-UG transition. Carefully consideration were given in choosing the required software for the thesis. The following software's were used for the OP-UG transition:

- Datamine;
- Isatis;
- Whittle; and
- XPAC.

Datamine software has been used widely in the generation of geological block models with the accuracy it requires. Its use in the manipulations of block models as well as to generate open pit designs for strategic mine planning have not been challenged. The two most widely software which uses LG algorithm to produce nested pit shells for ultimate designs are the Whittle and NPV scheduler. These two software are among the industry standard to generate optimum pit limits for pit designs. The choice of XPAC as the scheduling software for the OP-UG transition was based on license availability and the researcher's experience in using it. There were no issues of uncertainty in the use of these standard software packages for the OP – UG transition since the process is done in stages with each stage checked. Moreover, these software's have been found to produce similar results when applied correctly.

3.9 Conceptual OP-UG transition model

Decision-making is usually based on company values hence good decisions should be based on well-defined numbers, tools and experience. Transition indicators have both qualitative and quantitative checklists, used collectively to assist individuals in making the optimum decisions for their companies. Some of the qualitative transition indicators checklist for the OP-UG transition model are the following:

- Ore body properties and geometry;
- Required confidence in the resource model;
- Environmental, permitting and regulations, community issues rehabilitation requirements, air and water quality, and subsidence;
- Availability of appropriate workforce requirements including portability of open pit skills to underground;
- Availability of mining machinery, conversion of open pit machinery to underground as well as planning underground transition to coincide with surface equipment replacement time;
- Underground mining method selection;
- Geotechnical challenges including wall stability and crown pillars;
- Water issues, both groundwater and surface water; and
- Infrastructure requirements.

The modelling criteria for the OP-UG transition were based on the following quantitative transition indicators. These are presented in order of priority according to the weight they carry, but need to be used collectively:

- Margin as a ratio of gold price to cost; companies need high margins to operate in order to survive, sustain, and grow their businesses. Margins are made up of gold price used by the company to the cash cost. Total cash costs are calculated by adding cash operating costs (direct mining expenses, stripping and mine-development adjustments, third-party refining/transportation costs, and credits from by product sales if applicable) to royalties and production taxes. Cash costs do not include depreciation, depletion, and amortization, along with reclamation and mine-closure costs;
- Average grade at the run of mine (ROM) stockpile pad;
- Stripping ratio of the open pit mining: incremental stripping ratio of a pushback sequence applied to the unit mining cost;

- Net Present Value (NPV): net present value of the open pit (the underground and the pit combined or the underground alone). NPV financial returns using each method such as by generating future cash flows for each option; and
- Processed ounces.

The above Quantitative transition indicators formed the basis framework of the conceptual model for OP – UG transition, which could then be populated with data from the case study mines. However, the use of these transition indicators to make the OP-UG transition decision must take into account the following considerations:

1. The type of deposit, since different ore bodies behave differently with indicators.
2. The transition indicators are collectively used in making the optimum decision.
3. The quantitative indicators to be used, as rules of thumb, are the margin, grade and stripping ratio. This was followed by cost, NPV and the processed ounces, in addition to the qualitative indicators.
4. Grade uncertainty was cated for by the use of simulated models, while other uncertainties such as geotechnical issues were catered for by the use of qualitative indicators. Gold price to cost ratio corrected the uncertainty in gold price and cost.

3.10 Guide for OP-UG transition for gold mines and how to incorporate geological uncertainty in the transition

The following processes were suggested as a guide for OP-UG transition for gold mines to incorporate geological uncertainty in the transition:

1. Request and obtain geological block models or simulated models from the resource geologist with the handover documents.
2. Create simulated models, if not already created by the Resource geologist, with one of the known methods following laid down principles as outlined in the steps for creating simulated models.
3. Prepare the simulated models for optimisation using the macros and the set of assumptions and export the results and the shells with revenue factor of 1 for both OP and UG into Datamine.
4. Create mineable stopes for the underground and add the model to the open pit simulated models.

5. Using site data run a realistic mining schedule based on the mining capacities and plant throughput.
6. Evaluate the materials separately in each option (Option 1- open pit alone, Option 2- both open pit and underground and Option 3-underground alone)
7. Export the XPAC results for all the options into the Excel financial report developed to calculate the transition indicators for the deposit using the appropriate cost for the mine.
8. Draw the cumulative distribution and histogram graphs for analysis and comparisons of the results.

3.11 Chapter summary

This chapter described the modelling processes adopted for the conceptual OP-UG transition model. Each of the four case study mines was evaluated using the above steps. The Qualitative transition indicators were used to initially characterise each case study mine according to its geology as will be shown in the transition model in Chapter 5. After characterisation, the checklists were applied to each case study mine. Lastly, the Quantitative transition indicators were generated for each mine and evaluated against SDGM baseline values to inform the transition decision. These transition indicators were stripping ratio, grade, gold price to cost ratio, NPV and processed ounces. Chapter 4 describes the case study mines starting with Geita Gold Mine.

4.0 DESCRIPTION OF CASE STUDIES: OPEN PIT - UNDERGROUND TRANSITION

4.1 CASE STUDY 1: GEITA GOLD MINE

Geita Gold Mine (GGM) is wholly owned and managed by AngloGold Ashanti Limited. The deposit is mined as an open pit but with an underground potential. It is one of the mines looking at the possibilities of initiating a transition from open pit to underground and as such was used as a case study mine.

4.1.1 Location and background

The mine is situated about 4 km west of Geita town and about 120 km southwest of Mwanza in Tanzania. It is approximately 20 km to the southeast of Lake Victoria in the Mwanza Region of Tanzania. Figure 4-1 shows the location of Geita Gold Mine.

Figure 4-1: Map showing the location of Geita Gold Mine (Courtesy: GGM)

GGM is an opencast gold mine taking ore from a number of sources (pits and stockpiles). The operation is centred on the Nyankanga pit, which is the main production source. There are nine satellite pits at varying distances from the treatment plant. The Nyankanga pit is being mined in a number of pushbacks that require significant waste stripping. The various pits produce approximately 6 million tonnes per annum (Mtpa) from the various sources at an average grade of approximately 4.0 g/t. In 2011 the mine had 4.73Moz of Mineral Reserves. In 2011 there were 3541 employees (including 1820 contractors). Table 4-1 shows the mining fleet while Table 4-2 shows the mining statistics of the GGM. Figure 4-2 shows the Nyankanga pit with the mine infrastructure. Figure 4-3 shows the stripping ratios for the mine for the period 2007–2020.

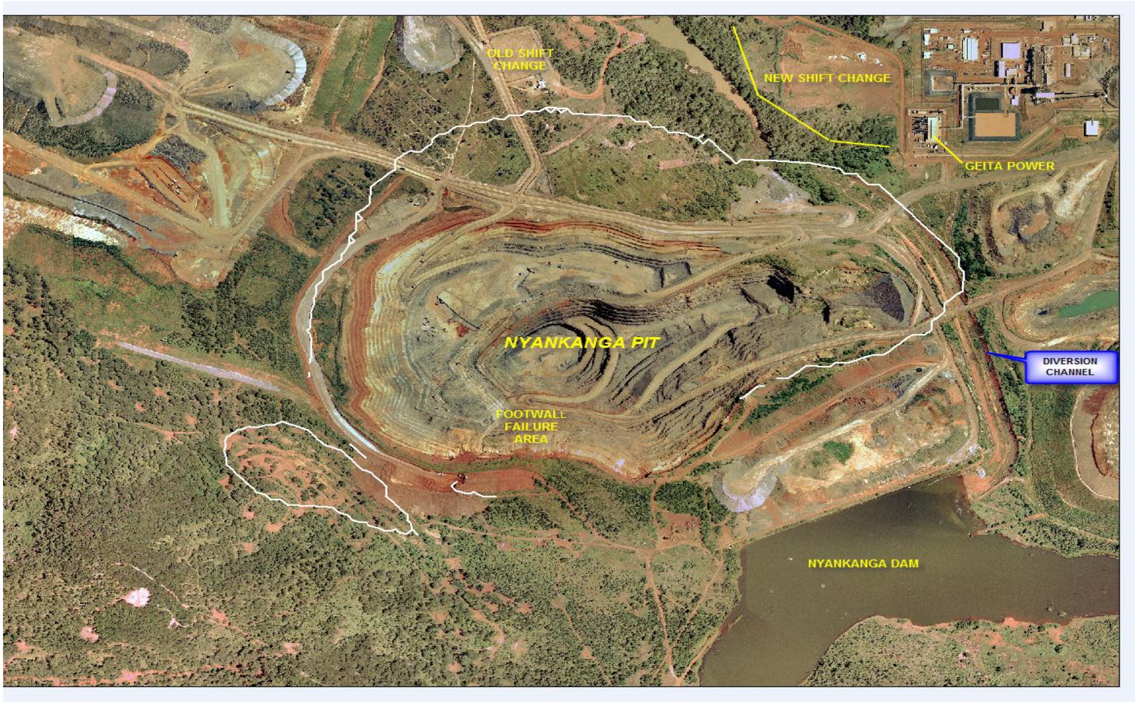

Figure 4-2: Location of Nyankanga pit and mine infrastructure at Geita Mine (Courtesy: GGM)

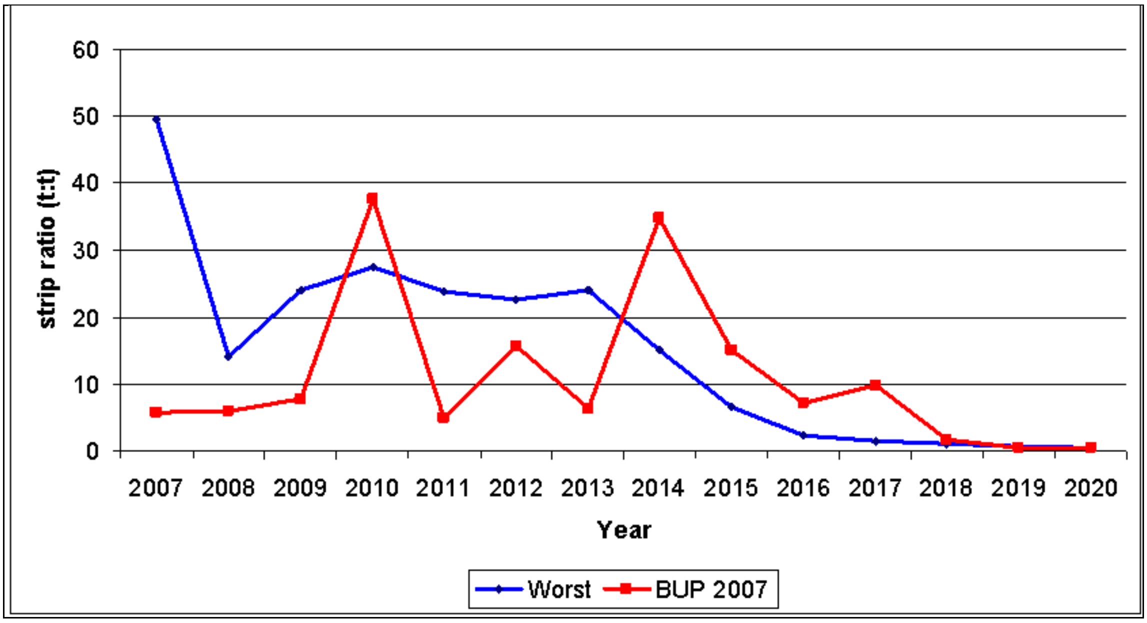

Figure 4-3: Nyankanga stripping ratio (Courtesy: GGM)

Table 4-1: Open pit mining fleet for Geita Mine

Equipment	Number
Liebherr 994 Hydraulic Excavators	4
Komatsu 1800 Hydraulic Excavators	4
Caterpillar 5130 Hydraulic Excavator	1
Komatsu PC 1100 Hydraulic Excavators	3
Komatsu HD785 -240t dump trucks	36
Caterpillar 777 – 90t dump trucks	15
Tamrock 1100 Blast hole drill rigs	2
Atlas Copco RocL8 Blast hole rigs	13
Terex MT 4400 AC Drive 240t dump trucks	4
O&K RH 340 Hydraulic Excavator	1
Associated service units-dozers, graders and water carts	1

Table 4-2: Geita mining statistics

Item	Quantity or Description
Resource	12.2 Moz
Reserve	6.5 Moz
Average LOM Annual Production	650 Koz Average
Mill Capacities	6 Mtpa
Producing Pits	Three increasing to five in 2008
Life of Mine	20 years from 2008
Projects in Resource	10 Ore Pits and planned underground

4.1.2 History of Geita Gold Mine

Historical mining in the area has taken place for many years, from the 1930s through to 1960s and produced almost 1 Moz of gold. On-going small-scale mining continues to this day. Table 4-3 shows the list of events at GGM in chronological order.

Table 4-3: Chronological list of events at Geita Gold Mine (GGM)

Year	Event
1896	Gold discovered in the Geita district
1934	Gold was first discovered at Geita
1936 -1966	Old Geita mine produced about 1 million ounces at 5.3 g/t
1966	Old Geita mine closed due to gold price and political changes
1994	Cluff Resources started exploration
1996	Ashanti acquired Cluff Resources
1998	Ashanti acquired Samax
1999	Nyankanga open pit commenced in August 1999
2000	AngloGold acquired 50% of Ashanti's Geita Gold Mine
2000	Plant commissioned at 4.2 million tonnes per year and in June 2000 first ounce poured by new plant
2002	GGM produces its 2 millionth ounce
2004	AngloGold and Ashanti merged to form AngloGold Ashanti Ltd
2005	GGM produced its 3 millionth ounce
2005	GGM commenced Owner Mining Operations

4.1.3 Geology and ore body properties

The total known strike length of the mineralized trend is 4 km to 5 km. The dip varies between 22° to 55° at Geita Hill. Geita orebody extends from an average width of 30 m and a strike direction of 125° (north-west to southeast). The strike length considered for mining is 1450 m. The ore body extends from 100 m below surface to approximately 435 m below surface. Currently economic mineralisation occurs, in places, across a 15km strike from Nyamulilima in the east to Kukuluma in the west. Geita is characterized by high grade ore feed from the Nyankanga pit with supplementary feed from the Geita Hill pit and nine other satellite pits. Mining operations remain suspended at Kukuluma and Matandani due to the nature of the ore. Geita trend is a 5 km mineralised structure trending WSW – ENE and dipping 40^{o} to 60^{o} N to NW. It obliquely crosscuts stratigraphy at a shallow angle and hosts more than 70% of Geita's known Mineral Resources. GGM is serviced by a 5Mt per annum carbon–in-leach (CIL) processing plant. Artisanal mining is still existent at various parts of the operation such as Geita Hill East Pit, Star and Comet and Roberts satellite pits.

The geological model, which was used for this study, was completed in mid-June 2010 and handed over to the resource evaluation team for further evaluation and estimation. The initial resource models for Nyankanga were estimated using original exploration reverse circulation and diamond drilling. The Nyankanga deposit forms the southwest limit of current known resources along the Geita trend and sub crops in low ground below 10-15 m of barren, transported laterite cover. The main ore body ranges up to 50 m thick in the central part of the deposit and dips sub-parallel to stratigraphy. The strike follows that of the stratigraphy and numerous steeper mineralised structures up to 10m thick occur as imbricate splays in the hanging wall. Grade distribution within the ore body is primarily controlled by lithology and structure. Areas of high grade generally represent uniformly mineralised Banded Iron Formation (BIF) with grades up to 20 g/t spread over a thickness of 10 m. In these areas, the ore body is wider but has a more erratic gold distribution and a lower average grade.

The different estimation zones are also shown in Figure 4-4. The main Zone 1 (MZONE 1) comprises of a higher grade. As observed in the geological model the diorites are generally lower grade while the BIF comprise of higher-grade lenses. For this reason, the Branded Iron Formations were estimated separately from the diorite units and later combined into a common block model that forms Zone 2 (MZONE 2). Exploration drilling is ongoing with efforts currently directed at increasing confidence in the Nyankanga ore body, and in the Nyankanga underground mining targets. The estimation methodology has evolved since 2005 to embrace non-linear techniques. Updates on resource models have shown that the current estimation method (Uniform conditioning) is suitable for the ore bodies at Geita Gold Mine. Figure 4-5 shows Nyankanga orebody domains subdivisions and Figure 4-6 shows the Geita structural trends.

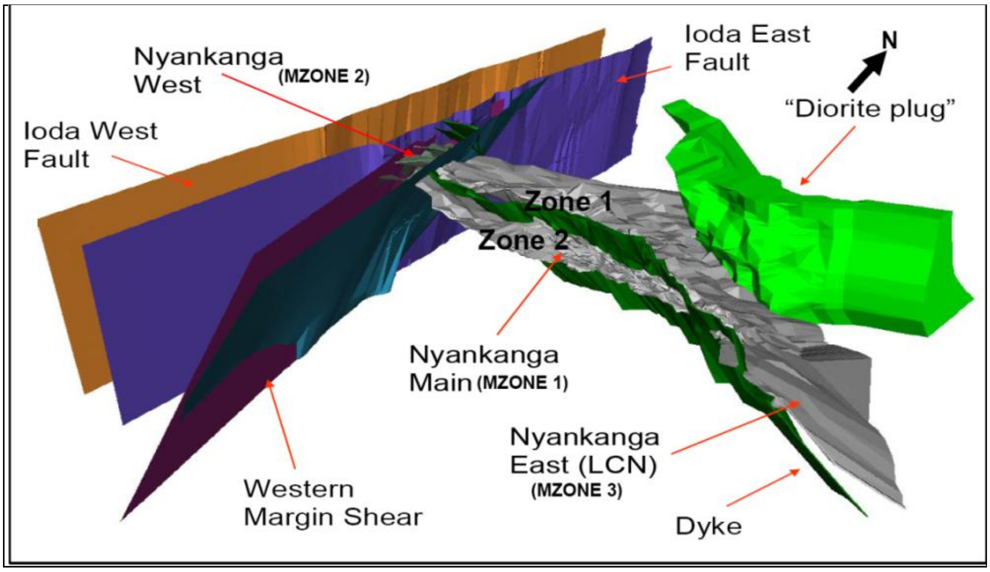

Figure 4-4: Nyankanga ore body showing mineralised zones (Courtesy: GGM)

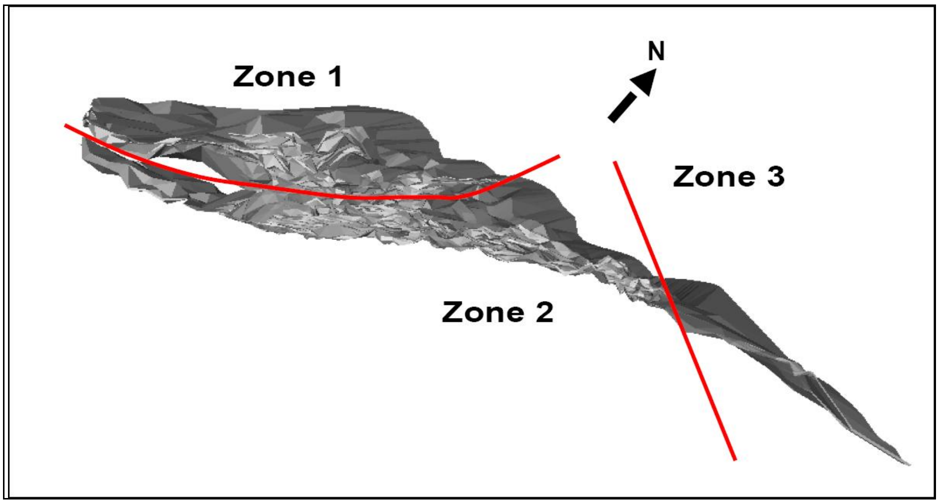

Figure 4-5: Geita domains subdivisions (Courtesy: GGM)

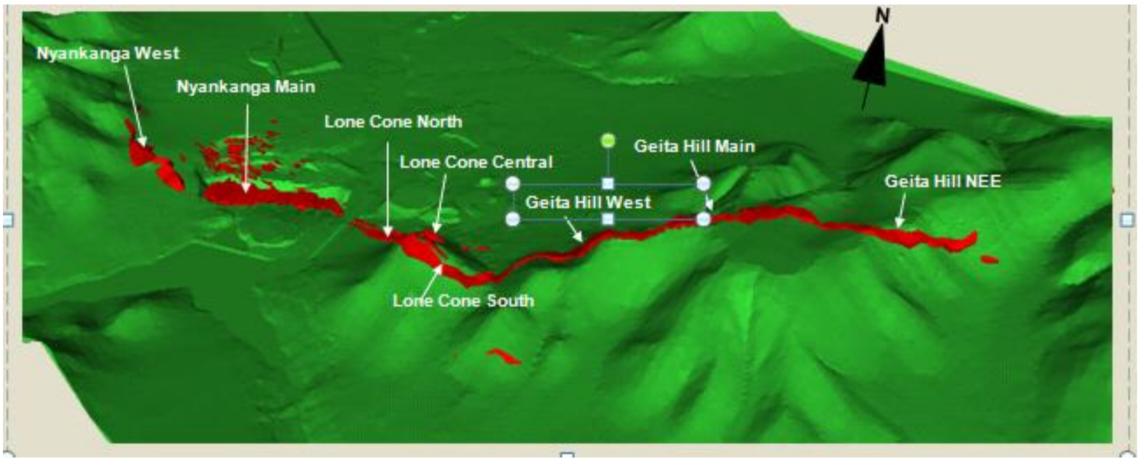

Figure 4-6: Geita structural trends (Courtesy: GGM)

4.1.4 Transition plans

The planned pit depth of the Nyankanga pit is 380 m below the surface. The Nyankanga pit and its pushbacks remain the backbone of the mine. The pit has ore of the higher grade areas in the mine. The ore supply from this pit flows constantly to avoid an ounce gap in the mine's life. In addition to the open pit there is an opportunity for underground mining where ore bodies extend below the existing open pits shells, particularly at Nyankanga, Geita Hill, Ridge 8 (Nyamulilima area), Kukuluma and Matandani pits. It is planned in future, to mine these deposits by underground means on completion of the open pits, if economical. In 2000 to 2001 after exploration work drilled the deeper portion of the Nyankanga deposit, studies were done to assess the underground potential. The study focused only on the economic viability of the underground potential.

In February 2004; AMC Consultants completed a study on Nyankanga underground potential, which was included into the Geita Business plan 2005 as a level 3 plan. Table 4-4 shows the project level classification as used in AngloGold Ashanti Limited. In June 2007 Turgis Consulting completed the study at a pre-feasibility level for underground potential below the Nyankanga, Ridge 8 and Geita Hill pits. In November 2007 Turgis Consulting completed the study for the underground potential at Nyankanga, Geita Hill and Ridge 8 and this was included in Business plan 2008 at level 3(a). The conceptual study showed less upside potential mostly due to lower recoveries, cost escalation and higher plant costs. Additional upside "pre-resource" underground potential from Nyankanga, Geita Hill, Kukuluma and Matandani was included into the 2009 business plan at level (3b). Potential trade-offs for Nyankanga underground were considered, designed and evaluated. Such trade off studies included examining alternative mining methods, rock handling declines versus vertical hoisting and belts or incline rails out of the pit. In addition, there was a need to investigate the possibility of mining more selectively to improve grades.

Table 4-4: Project level classification (Courtesy: AngloGold Ashanti Limited)

Level	Resource Category	Non Project / Project	Reserve Yes / No	Guideline
1	M & I	Non Project	Yes	Everything is mineable from current infrastructure with no further project capital investment needed, i.e. only ORD and SIBC.
	Inferred	Non Project	No	As above, and must be contained within LOM pit shell or design.
	M & I	Projects (Approved)	Yes	Current board approved project in execution.
	Inferred	Projects (Approved)	No	Part of approved project.
= Level 1 Base Plan - this is the plan against which new projects are judged, i.e. projects must compete to improve the Base Plan				
+ 2a	M & I	Project	Yes	Project at an advanced stage of study, i.e. a completed and reviewed pre-feasibility with single option outcome and/or test work, by year end. Acceptable financial outcome (NPV > 0), permission to proceed to full feasibility, and board intent to proceed with project on successful completion of feasibility study. The payback period must be covered by probable reserves.
	Inferred	Project	No	Inferred resources will generally be < 50% of the above project and must demonstrate adequate financial provisions and a plan to convert inferred to indicated through exploration within the payback period.
+ 2b	M & I & Inferred	Project	No	Project at an advanced state of study but has no chance of completing a pre-feasibility with single option outcome and/or test work, by year end. The payback period is not yet covered by probable reserves. Inferred resources will generally be > 50% of the above project but must demonstrate adequate financial provisions and a plan to convert inferred to indicated through exploration within the payback period.
= Level 2 Plan – this is the plan against which Reserves are declared, and includes approved projects containing M & I				
+ 3a	M & I & Inferred	Non Project	No	Should be little or no M & I ; mostly inferred blocks, contains the old Level 2b non-projects.
	M & I & Inferred	Project	No	Project at scoping or conceptual level, and/or M & I << 50%, i.e. projects at a low level of geological confidence and/or engineering work.
+ 3b	BST & BSI	Non Project	No	Ground is accessible from current infrastructure.
	BST & BSI	Project	No	Project is at scoping or conceptual level.
= Level 3 Plan - this is the plan which infers growth toward the maximum value of the asset and is used for Impairment calculations				

4.2 CASE STUDY 2: CERRO VANGUARDIA SA MINE

Cerro Vanguardia S.A (CVSA) is an open pit mine north-west of Puerto San Julian in the province of Santa Cruz in Argentina. The ownership of the mine comprises of AngloGold Ashanti Limited (AGA) with 92.5% and the Fomicruz (the province of Santa Cruz), with 7.5%. In the year 2002, Bajo de la Alumbrera and Cerro Vanguardia together produced about 94% of Argentina's gold and were significant producers of silver. AGA obtained the 92.5% interest in the Cerro Vanguardia mine following the acquisition of an additional 46.25% interest in July 2002. It is the only mine owned by

AGA in Argentina and among one of the five mines operated by AGA in the America's region. The other AGA mines in the region are the Cuiaba complex, *Corrego do Sito* complex, MSG and CC & V mines. In 2011, CVSA produced gold of 196,000oz, and 2.7 Moz of silver as a by-product.

4.2.1 Location and background

CVSA mine is located about 120 km northwest of Puerto San Juliàn in the province of Santa Cruz, Southern Argentina. It is 195 km from Puerto Deseado. The mine is accessible by plane from Buenos Aires to Comodoro Rivadavia or Rio Gallegos and then by road to the mine site. CVSA is approximately 650 km and 540 km from Comodoro Rivadavia and Rio Gallegos, respectively. The mine is located on a relatively flat plateau with an average annual rainfall of 200 mm and with average temperatures of 13ºC in summer and 3ºC in winter. The vegetation is well outside the tropics. Figure 4-7 show the location of CVSA mine while and Figure 4-8 show the location relative to nearby towns. Figure 4-9 shows the CVSA mine infrastructure.

Figure 4-7: Map showing the location of CVSA Mine (Courtesy: AGA)

Figure 4-8: Location of CVSA Mine relative to nearby towns (www:argentina.gov.ar)

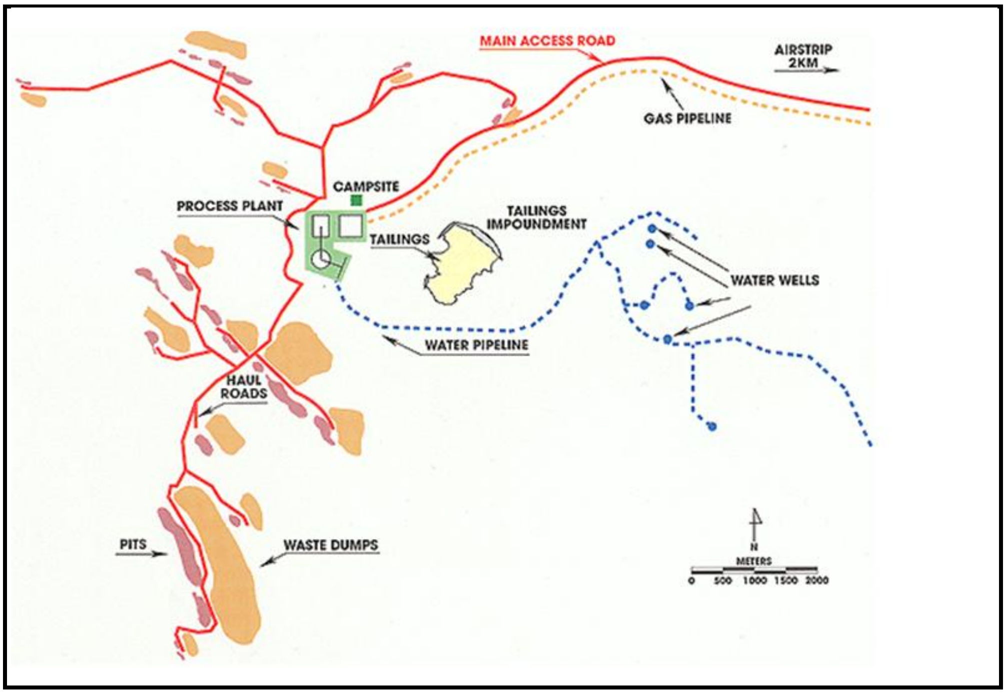

Figure 4-9: Site plan of CVSA (Courtesy: AGA)

The planned pit depth of the pit (Loma del Murto) is 96 m below surface. CVSA mine employs 1065 permanent employees with a further 579 people working for contractors as at 2011. CVSA consists of multiple small open pits. The plant processed a total tonnage of about 950,000 tonnes per year although it has a maximum capacity of 1,000,000 tonnes per year. In the geological block models used for production planning, tonnages are estimated taking into account *in situ* humidity and dilution was considered at 1 m (0.5 m each side of the ore body). Due to the nature of the ore body, the pits are designed with a ramp of 12 m increasing to 16 m. Bigger berms are left at every 300 to 400 m to allow for crossing of trucks. All pits have a ramp gradient of 10%. The mine started generating simulated model for geological risk quantification in 2004. CVSA processing unit has a mill, heap leach and underground mine producing about 220 000 ounces per year. Table 4-5 shows the mining fleet for CVSA while Figure 4-10 shows the LOM stripping ratio for CVSA for the years 2009 to 2018.

Table 4-5: CVSA mining fleet

Type	Make	Quantity	Capacity
Haul Trucks	Cat 773D	5	50 t
	Cat 773E	8	50 t
	Cat 773F	3	50 t
	Cat 777D	4	90 t
Loaders and Excavators	Cat 988G	1	6 m^3
	Cat 988H	3	6 m^3
	Cat 992G	3	11.5 m^3
	Cat 385B	1	5 m^3
	Cat 385C	1	5 m^3
Blasthole Drills	Tamrock Pantera 1500	8	5 "

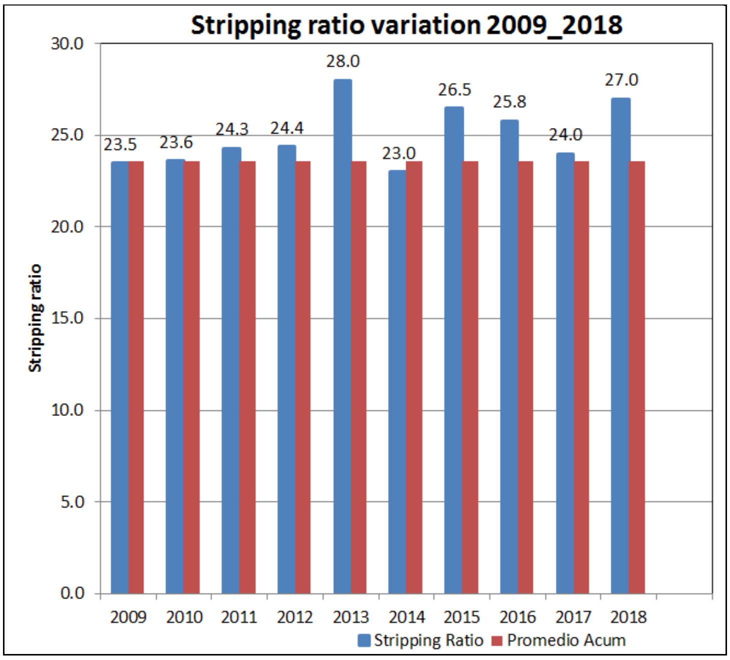

Figure 4-10: Stripping ratio variation for CVSA

4.2.2 History of **CVSA Mine**

CVSA is one of the low-cost gold producers within AngloGold Ashanti's group that is able to meet their operational targets. Table 4-6 shows a list of events at CVSA mine in chronological order.

Table 4-6: Chronological list of events at CVSA Mine

Year	**Event**
1976	Exploring for barytes in Santa Cruz province
1987	50/50 mineral exploration joint venture with AMSA, Anglo American's South American holding company
1991	Mincorp identified the potential at Cerro Vanguardia and secured the rights to a 514km² concession
1992	Mincorp exploration and metallurgical test work
1996	Cerro Vanguardia SA was formed
1997	Construction work started
1998	First gold was shipped
1998–99	AngloGold acquired the 46.25% group interest
2004	AngloGold merged with Ashanti Goldfields to form AngloGold Ashanti

4.2.3 CVSA geology and transition **plans**

The CVSA district represents one of the most extensive epithermal quartz veins in the world; comprising over 205 km, long quartz vein outcrops. In the southern third of the property (Lazo), the veins strike at 10ºW. In the central portion (Cerro Vanguardia Hill), the veins trend N45ºW and dip 70ºNE. Most veins dip steeply at 90º to 70º. The vein dimensions range between 150 m to 11 km along the strike and from 0.5 m up to 10 m thickness, averaging about 3.5 m. Mineralised zones have variable extensions from 150 m to 2 200 m and the vein net width is from 2 cm to 10 cm in thickness.

Gold and silver mineralisation at Cerro Vanguardia occurs within a vertical range of about 150 m to 200 m in a series of narrow, banded quartz veins that occupy structures within the Chon Aike ignimbrites. These veins form a typical structural pattern related to major north south (Concepcion) and east west (Vanguardia) shears. Figure 4-11 shows a 3D view of CVSA model used for the OP-UG transition.

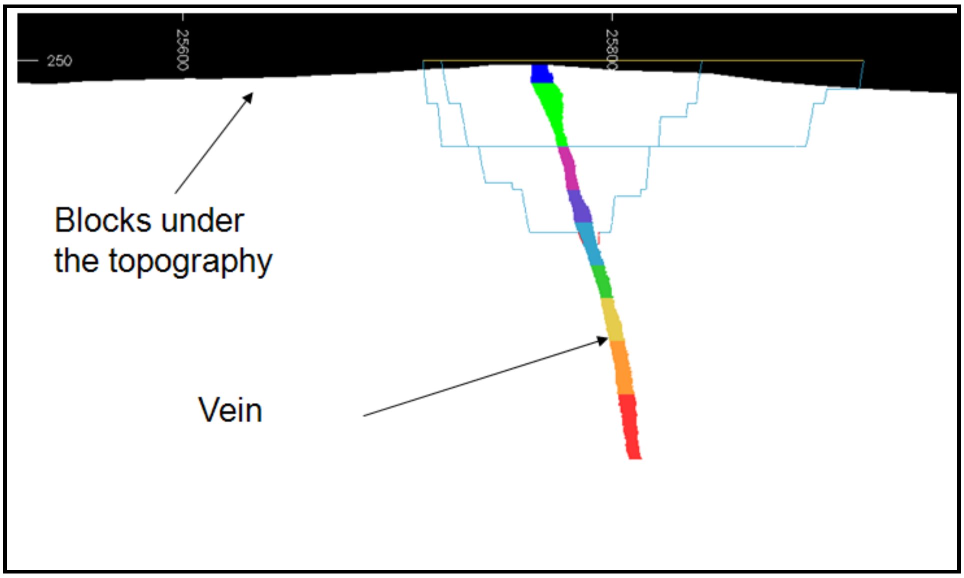

Figure 4-11: 3D view of CVSA model for the project (Courtesy: CVSA Mine)

4.3 CASE STUDY 3: SADIOLA GOLD MINE

Sadiola Gold Mine (SGM) is mined by the *Société d'Exploitation des Mines d'Or de Sadiola S.A*. (SEMOS), the operating company formed through a joint venture between AngloGold Ashanti Limited (AGA, 41%), IAMGOLD (41%) and the Malian Government (18%). Mining activities take place in five open pits (the Sadiola Main Pit, FE3 pits, FE4 pits, Tambali and Sekokoto). The Main Pit deposit comprises of the oxide portion and the deep sulphides constitute the unweathered material below the pit. Mining of the oxide portion of the main pit finished in 2004 and the unmined deep sulphides were used for the research study.

4.3.1 Location and background

The Sadiola deposit is located in the north-west of Mali, about 77 km to the south of the regional capital of Kayes. The Republic of Mali is a nation bordering Algeria in the north, Niger in the east, Burkina Faso and the Côte d'Ivoire in the south, Guinea in the southwest and Senegal and Mauritania in the west. The country has a population of almost 12 million people. A regional gravel road to Kayes can be used to access the mine site. Kayes is serviced by rail, road and air from Bamako and from Dakar, the capital of Senegal. The climate of the region can be described as a tropical climate with temperatures ranging from 27°C in December, up to 33°C in May. Annual rainfall averages 750 mm with the majority of this falling between April and October. The mine site is serviced regular flights from Bamako and Dakar to the site's airstrip. Figure 4-12 shows the location of Sadiola Gold Mine.

Figure 4-12: Map showing location of Sadiola Mine (Courtesy: AGA)

4.3.2 History

Locals initially identified the Sadiola deposit as evidenced by the widespread artisanal gold workings and small-scale mining. From 1987 to 1989, a large regional geochemical survey was carried out for the government of Mali as part of an aid program financed by the European Development Fund. The survey identified high gold, arsenic and antimony anomalies near the villages of Sadiola and Dinguilou. In 1990, the government of Mali granted exploration rights in respect of the Sadiola area to an entity associated with the initial formation of IAMGOLD. Subsequent geological mapping, geophysical surveys, pitting and core drilling identified a significant oxide gold deposit.

In December 1992, WGM estimated a probable reserve of 22.3 million tonnes of oxide mineralization grading 3.3 g/t gold. Later in 1992, IAMGOLD negotiated a joint venture agreement with Anglo-American for the development of the Sadiola mine. The gold assets of Anglo-American were merged to form AngloGold Ashanti who are currently the operator of the Sadiola Gold Mine. Table 4-7 shows a list of events as at Sadiola Gold Mine in chronological order.

Table 4-7: List of chronological events at Sadiola Mine

Year	Activity	By
From 1987 to 1989	Regional geochemical survey	Government of Mali
1990	Granted exploration rights/formed company	IAMGOLD
1991 and 1992	Large exploration programme	Watts, Griffis and McOuat
1992	JV agreement with Anglo-American	IAMGOLD
December 1992	Estimated Probable Reserve of 22.3 million tonnes of oxide mineralization grading 3.3 g/t gold.	Watts, Griffis and McOuat

4.3.3 Geology, current plans and production

The mineralized zone dips from east to west from 70° to vertical and plunges to the south-west at 20°. The width of the ore body varies from 30 m to 70 m and extends to about 1,000 m on strike. The ore body plunges 20° to the south. The dip is about 70° to the vertical and the ore body extends over about 1 km. The thickness of the mineralization is about 40 m to 70 m. The resource to 250 m depth is classified as 'Indicated' and as 'Inferred' between 250 m and 500 m. The country rock on the eastern (footwall) side is marble (a non-foliated metamorphic rock type) and on the western (hanging wall) side, it is Meta-sandstone or greywacke (a clastic sedimentary rock type). Geologically, the Sadiola concession occurs within the Pre-cambrian Birimian System (2.17-2.18 Ga) of West Africa. The Sadiola deposit is found in this belt along with other deposits such as Yatela and the more southerly Loulo and Segala.

The Sadiola Hill deposit originally consisted of two zones, an upper oxidised cap and an underlying sulphide zone. From 1996 until 2002, shallow saprolite oxide ore was the primary ore source. Since 2002, the deeper saprolitic sulphide ore was mined, progressively replacing the depleted oxide material. Primary mineralisation at Sadiola is structurally controlled and deposited by hydrothermal alteration. The deposit occurs along the 10° striking Sadiola Fracture Zone ("SFZ") in the northern section of the Kenieba-Kedougou window. The SFZ is interpreted as a brittle-ductile splay off the

Senegal-Mali Shear Zone at a sinistral releasing bend. The SFZ follows the steeply west dipping contact between lithologies of the Kofi Formation and in particular, meta-greywacke to the west and impure meta-carbonate to the east. The SFZ and its wall rocks are intruded by discontinuous diorite dykes, which may contain a weak mineral foliation and rarely intense ductile deformation. A summary of the domains are presented below, followed by an explanation on how these domains were derived. A zone code (KZONE) was assigned to each domain. A W-E section illustrating the major lithologies in the area is presented in Figure 4-13.

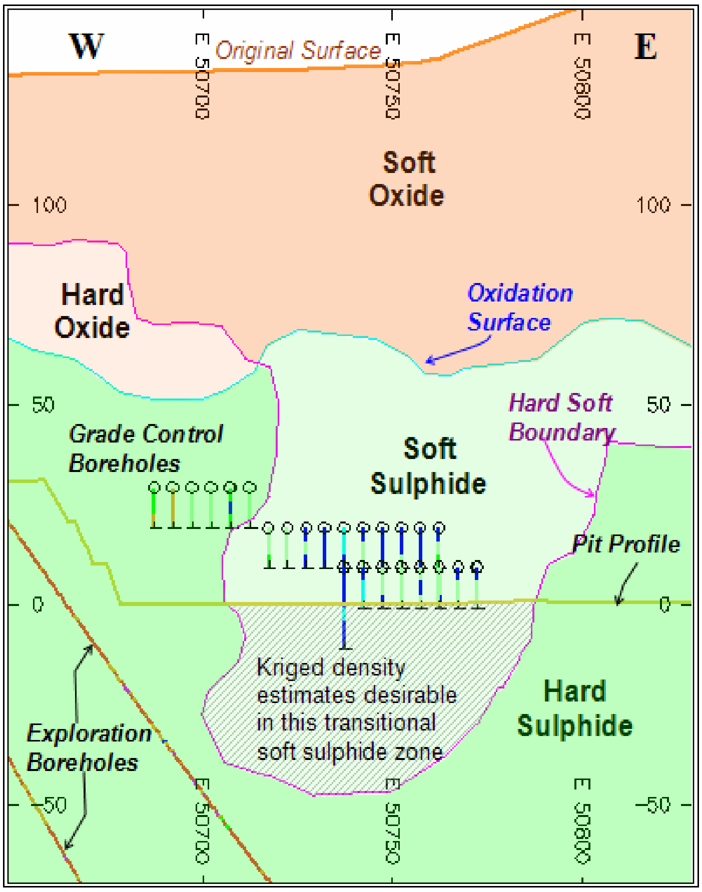

Figure 4-13: W-E section showing the various material types (Courtesy: AGA)

The wireframe envelopes were used to select the samples for estimation. Samples within the wireframes were classified as mineralised and all samples outside of the wireframes were classified as un-mineralised (waste). Appropriate domain codes were assigned to the samples, which are summarised below. Table 4-8 shows the mining fleet in Sadiola.

Table 4-8: Sadiola mining fleet

Equipment		Number of Units in Operation	Average engine hrs/model*	2008		2009 **		2010	
Type	Model			Rebuild	Replace	Rebuild	Replace	Rebuild	Replace
Excavator	O&K RH120 E	2	12000			1			
	O&K RH120 C	1	12000						
	O&K RH 40	2*	12000						
	CAT 330	1	12000			1			
Trucks	CAT 777 C	8	18000	3		1			
	CAT 777 D	15	18000	10		1			
	CAT 773 D	13*	18000	2					
FEL	CAT 992 G	2	18000						
	CAT 998 F	1	15000	1					
Dozers	CAT D9R	4	15000	1					
	CAT D8N	2*	15000	1					
	CAT 824 G	1	15000	1		1			
Graders	CAT 16 H	1	20000			1			
	CAT 14 H	3*	20000	1					

4.3.4 Transition plans

The current Sadiola main pit finished mining towards the end of 2004. The planned pit depth of the Sadiola main pit was 220 m below the surface. The pit was originally constructed using data that was primarily acquired in the oxide area. At Sadiola, a substantial, partially refractory, primary sulphide resource that remains open along strike and at depth was delineated beneath the main pit oxide deposit. Studies commenced in 1999, with further investigations in 2001, 2003 and 2005. Exploration for the deep sulphides and open pit mine designs, metallurgical test work and preliminary processing flow sheets were developed. Due to the hardness and refractory nature of the deep sulphides, the Sadiola treatment plant infrastructure is not suitable and significant changes to the metallurgical treatment plant would be required.

In September 2007, Turgis Consulting were commissioned to develop a pre-conceptual design and costing of an underground mine below the last pit shell. Key assumptions in the study were:

- The boundary between the open pit and underground potential was assumed as the fully depleted pit.
- Power generation will be hydro-electrical at a cost of USD 0.01 per kWh; and
- The ore would be treated using the heap biox process.

4.4 CASE STUDY 4: MORILA GOLD MINE

The Morila deposit occurs within the 200 km Morila Lease and is owned by Morila SA, a Malian registered company created by Randgold Resources Limited. The Morila shareholding comprises AngloGold, Randgold Resources and the Malian Government with 40%, 40% and 20% respectively.

4.4.1 Location and background

Morila Gold Mine is located in the Sikasso region in southern Mali, approximately 280 km by road south-east of the capital city, Bamako. Figure 4-14 shows the location of Morila Mine in Mali.

Figure 4-14: Map showing the location of Morila mine (Courtesy: AGA)

The planned pit depth for the Moria was 252 m below the surface. The mining at Morila was outsourced to Somadex, a French contract mining company that was wholly owned by DTP Terrassement. The mining contractor manages both the drill and blast and load and hauls operations at Morila, with the short and long term planning, survey and mineral resource management functions being undertaken by Morila S.A. The annual mining capacity was estimated at 9.7 million BCM (25.4 million tonnes). The final pit will have a length of 1,190 m, a width of 820 m with the pit floor 196 m below the surface. The life of mine pit surface area equates to 66.6 hectares and on average 6.4 million cubic metres per annum (17.2 million tonnes) will have been mined over the LoM. The life of mine stripping ratio was approximately 3.7:1. Run-of-Mine (ROM) ore was transported directly from the open pit in 90 tonne payload, CAT 777D haul trucks.

This material was either stockpiled on the ROM pad or directly tipped into the primary crusher. The Nordberg-54/75 primary crusher reduces the ore down to less than approximately 300 millimetres in a single stage, open circuit. Production started on the 4th of October 2000 with first gold being poured on the 18th of the same month.

4.4.2 History

Exploration was conducted in the area since the early 1950’s by French, Belgian, Russian and Malian companies. Focused systematic regional exploration of the area began in the mid 1980’s. Soil anomalies were followed up in the early 1990’s by BHP Billiton through limited diamond drilling which intersected ore grade mineralisation. Exploration in the Morila area was discontinued when BHP made a strategic decision to disinvest from Mali. Subsequent acquisition of the permit by Randgold in the late 1990s resulted in renewed exploration activity. Randgold Resources Limited (RRL) acquired the Morila permit when they acquired BHP Minerals in Mali in October 1996. In June 2000, successful joint venture negotiations between Randgold Resources and AngloGold resulted in the acquisition of a 40% portion, including operational management of Morila by AngloGold. Table 4-9 shows a list of events, which took place at Morila Gold Mine.

Table 4-9: List of events at Morila Gold Mine

Year or Date	Activity	By
1950	Exploration conducted	French, Belgian, Russian and Malian companies
1980	Regional exploration	
1990	Acquisition of the permit	Randgold
1999	11 year life gold mine initiated	Randgold
4th October 2000	Commissioning of the plant	Randgold
16th October 2000	first gold was poured	Randgold
20th May 2009	Open pit operations ceased	Randgold

4.4.3 Geology, current plans and production

The Morila ore body is hosted within an interpreted metamorphosed impure arkose and feldspathic arenite (formerly recorded as metagreywacke), a metamorphic rock dominated by quartz, plagioclase, biotite and alkali feldspar. X-ray diffraction and petrology studies indicated a near uniformity of gangue (silicate) mineralogy throughout the sequence, extending several hundred metres beyond the proposed final pit depth. The south-eastern portion of the Morila pit incorporates a portion of intrusive tonalite, which is also uniform in composition, consisting of plagioclase, quartz and biotite with minor chlorite and amphibole (Weedon, 2004). According to Reynolds (1999), “the gold mineralization is hydrothermal in origin and is contained within altered meta-sediments close to the contact with intrusive tonalite. Alteration is commonly silica-feldspar alteration as well as minor argillic alteration. The visible sulphide mineralization consists of arsenopyrite, pyrrhotite, pyrite and trace chalcopyrite. Coarse visible gold is a common occurrence”. Figure 4-15 shows a section through the Morila ore body.

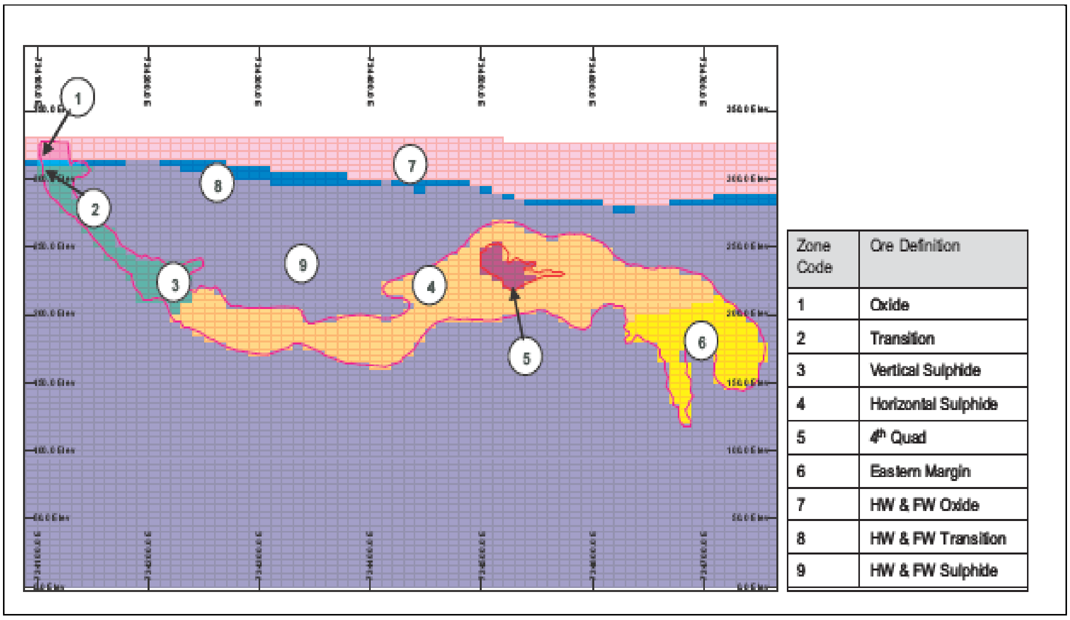

Figure 4-15: Section through Morila ore body (Courtesy: Morila Gold Mine)

The contractor’s primary loading fleet consists of a CAT 5130 shovel, two Liebherr 994 shovels, a Liebherr 994 excavator, a list of mining fleet currently in Morila is given in Table 4-10. The mined ore was processed through a conventional semi-autogenous, including a recycle crusher and carbon-in-leach (CIL) circuit together with a gravity gold recovery step with final residue reporting to a tailings storage facility. Table 4-11 shows the summary of the different rock types in the resource model. Table 4-12 shows the mining production statistics for Morila mine.

Table 4-10: Morila Mining fleet

Name	Type	Number
Excavators	Liebherr 994	1
Utility Excavator	Cat 385	1
Front End Loader	Cat 990	1
Haul Trucks	Cat 777and775	3
Graders	Cat 16G	1
Water Tankers	Cat 773	1
Track Dozers	Cat D10	1

Table 4-11: Summary of rock types in the resource model

ROCKTYPE	Description	Density	KZONE
Oxide	Above oxide/transitional wireframe	1.69	1 and 9
Transitional	Between oxide/transitional wireframe and transitional/sulphide contact	2.34	2 and 10
Sulphide	Below transitional/sulphide contact	2.78	3 to 7
Granodiorite	All Material within the Granodiorite wireframe	2.66	12
Tonalite	All Material within the Tonalite wireframe	2.66	13

Table 4-12: Mining production statistics

Description	Unit	2000	2001	2002	2003	2004
Total cash cost / tonne	US$/t	28.87	22.65	28.31	26.51	22.67
Total cash cost / ounce	US$/oz	99.27	102.5	99.15	109.06	176.1
Mining cost / tonne mined	US$/t	N/A	1.0393	1.0013	1.5211	1.6195
Mining cost /BCM mined	US$/BCM	N/A	2.1814	2.3703	4.0556	4.306
Stripping ratio		6.18	5.86	7.15	4.17	4.24

4.4.4 Transition plans

To establish that underground potential exists at Morila within the pit vicinity, a short scoping exercise was completed by creating an inventory model by only considering resources in life of mine pit shell. All other mineral resources were assumed to be mined. An external cut-off grade of 2.0 g/t was applied to the resources, which lie below the pit shell (pit5_may07tr/pt The results from the study show there are potential resources within the areas defined in the order of 1.5 Mt at a grade of 5.04 g/t producing 250 000 oz of gold at the cut-off grade of 2.0 g/t. The main concern was that the amount of material left below the pit was insufficient to support an underground operation alone. The block model provided relies on geostatistical analysis to divide the parent blocks (10m x 10m x 5m) into varying grade bands. However, closer examination of the ore body shown in Figure 4-16 revealed that the southern extremity was vertically too thick to extract using room and pillar without requiring fill. Given that fill would be required, it was considered that bench and fill mining would still allow selective mining to take place but would also be a more economical mining method.

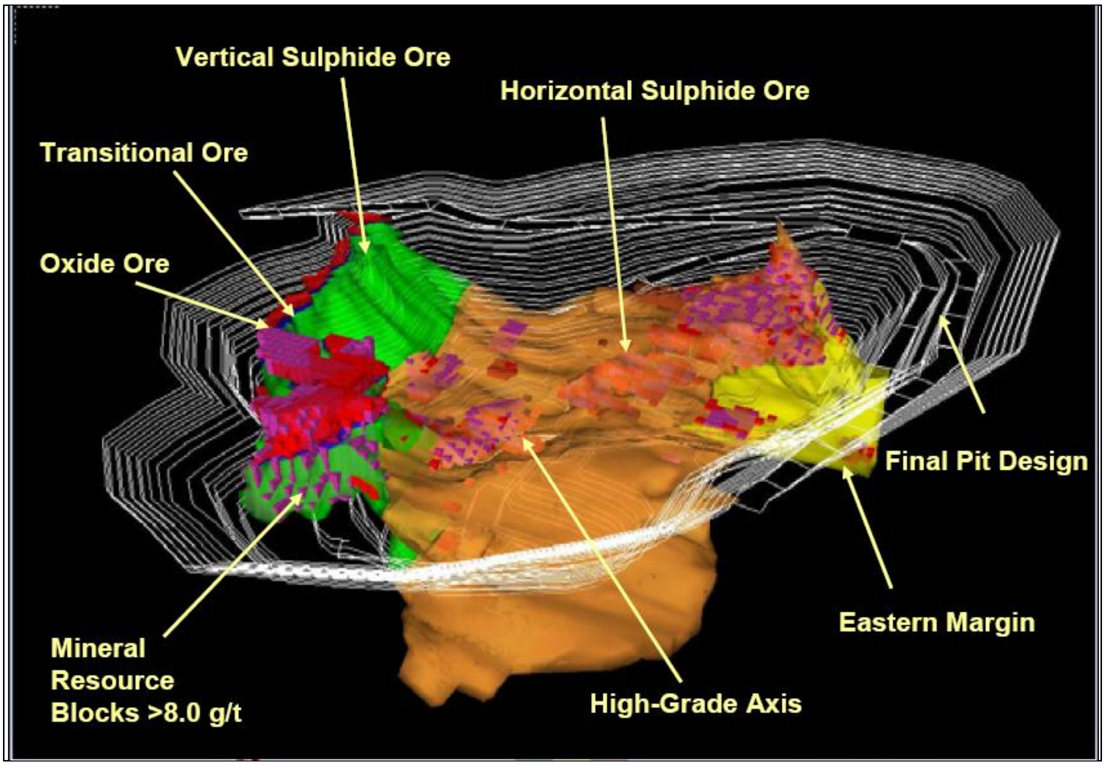

Figure 4-16: 3-D ore body with pit design (Courtesy: AGA)

As detailed earlier, the proposed mining method for both ore zones was bench and fill. The amount of stripping in advance of the stope was a function of geotechnical conditions as is the secondary roof support installed at this time. Stopes, as is the case in this mine, can be mined as primaries and secondaries, with mining starting at the base of the mine and progressing upwards working on top of fill. Production drilling of the stopes would be done using a downhole production drill rig. Loading and firing would also be done from the top stripped ore drive. Mucking of the stopes would via the bottom ore drive using remote operated front-end loaders. Material would trammed from the stope in loaders before being loaded onto trucks and hauled from the mine. If further drilling delineates more ore, an ore-pass system may become the most effective means of handling ore prior to truck loading. After mucking out the stope, backfill will be required. The span of stope that may be kept open will be determined by geotechnical analysis. Primary stopes will require either paste fill or hydraulic fill, the components of which can be sourced from the processing plant. Secondary stopes will be recovered with fill on two sides.

In March 2007, GijimaAST were commissioned to compile a pre-conceptual design, scheduling and costing for Near Pit Underground (NPU) potential at Morila. This culminated in a report dated 16 April 2007 and a brief presentation at the April 2007 Morila Limited Board Meeting, where it was agreed that no further drilling for Near Pit Underground (NPU) potential should be done. The average plant feed grade in the study was calculated to be 3.58 g/t at a gold price of $600/ oz and a grade of 3.7 g/t would be required to break even. The transition plan ended with a decision not to start an underground project since the material left below the pit is not enough to support an underground operation.

4.5 CHAPTER SUMMARY

The chapter has described the location, history and geology of the case study mines, namely Geita, CVSA, Sadiola and Morila. It also explained the transition decisions previously considered for each case study mine. The next chapter discusses Sunrise Dam Gold Mine located in Australia, which was used as a benchmark for the transition decision.

5.0 BASELINE VALUES FOR MODEL USING SUNRISE DAM GOLD MINE AS BENCHMARK

Sunrise Dam Gold Mine (SDGM) in Australia is 100% owned by AngloGold Ashanti Limited. The Sunrise deposit was discovered in 1992 and an initial resource was estimated based on predominantly oxide drilling. Sunrise Dam Gold Mine operations comprise a large open pit and an underground mine. The underground project commenced in 2004 and exploited the section of the Sunrise ore deposit that falls within the leases held by Acacia Resources Ltd.

5.1 Location and background

SDGM is located approximately 55 km to the south of Laverton. It is about 220 km northeast of Kalgoorlie in the Eastern Goldfields of Western Australia. The Sunrise Dam mine lies on the eastern shore of Lake Carey, some 770 km north-east of Perth. The Sunrise pit forms part of a much larger gold deposit, with known resources at depth currently being mined. The Sunrise section of the deposit, which straddles the title boundary, is within the adjacent leases of Placer (Granny Smith) Pty Ltd, a joint venture between Placer Dome Inc. (60%) and Delta Gold NL (40%). Figure 5-1 shows the location of SDGM.

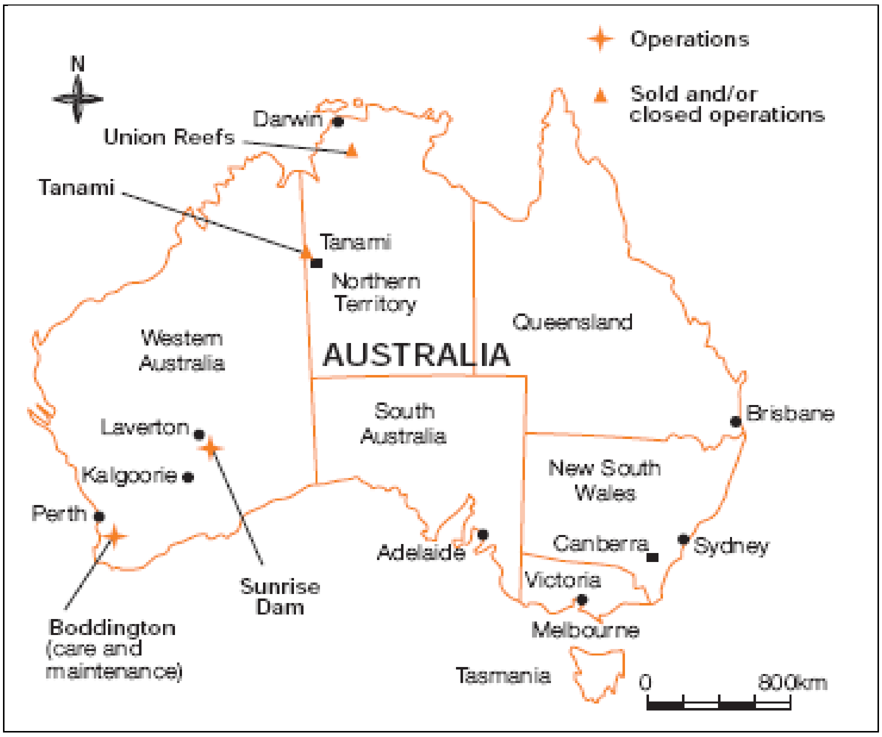

Figure 5-1: Location of Sunrise Dam Gold Mine (Courtesy: AGA)

The pits of the two operations overlap with Acacia pit mining mainly oxide ore, while the Placer (Granny Smith) operation is currently in primary ore. Acacia pit also operates the Cleo pit adjacent to the combined Sunrise pit. Mining development commenced in 1994 and ore production in the following year. The scoping study for the underground was completed in 2003 and underground development commenced in 2004. The underground project involved the development of two declines. Resource modelling was performed using two different types of estimation, conventional geostatistical estimation by ordinary kriging and conditional simulation performed for three of the geologic domains in the underground resource. Drilling at Sunrise Dam indicated that the sub-vertical high-grade zones that were a feature of open pit mining continued at depth. The transition enabled the underground potential for the ore body to be fully explored.

5.2 History

The ore body at Sunrise Dam is structurally and lithologically controlled within gently dipping high-strain shear zones. Host rocks include andesitic volcanic rocks, volcanogenic sediments and magnetic shales. The mine comprises a large open-pit operation and an underground project. Contractors carry out mining and ore is treated in a conventional gravity and leach process plant. Table 5-1 shows a chronological list of events at Sunrise Dam Gold Mine.

Table 5-1: Chronological list of events at Sunrise Dam Gold Mine

Year	Event
1988	Gold deposit discovered
1993	Discovery of Cleo main mineralised zone
1995	Gold mining operations started
1996	Feasibility completed
1997	First gold was poured
1999	AngloGold acquired the mine
2002	AngloGold acquired the Sunrise Lease
2004	Underground operations commenced
2006	Conversion of its diesel power generators to liquefied natural gas
2009	Mine produced 94,000 ounces of gold

5.3 The use of SDGM for benchmarking against mining industry

The underground mining in Sunrise Dam Gold Mine commenced in 2004. A contractor, BARMINCO, was used to undertake the UG mining with full complement of supervision, operation and maintenance staff on-site, and production equipment fleet. The mineable ore bodies in the SDGM underground occur in two primary geometries; low to moderately dipping veins and steeply dipping veins and fault structures. In the low-moderate dip ores, the limited ore thickness and projected production grades dictate partial extraction using two variations of room and pillar mining. The steeper ore bodies together with other factors favoured sublevel open stoping with backfill. Maintenance of the pit bottom dewatering necessitated the construction of a sump at the pit bottom. Production sequencing allowed the deposit to be mined vertically upward. This sequence was interrupted in specific locations because the crown pillar had to be recovered in the future. Recovery was delayed until the final periods of the LOM. Muck haulage over the LOM was by 50 tonne capacity haul trucks provided by the mining contractor due to the scattered character of the ore within the mine making the existing development not suitable for conveyor transportation.

For this purpose, Sunrise Dam Gold Mine (SDGM), which made the OP-UG transition in 2004, was used as the baseline mine. Analysis of SDGM with the transition indicators information was based on some key historical operating data from 1996 to 2002 production (beginning of the transition studies) as indicated in Table 5-2. The same transition indicators including the gold price were used for the other case study mines.

Table 5-2: SDGM 1996-2002 production summary

	Units	1996 - 1999	2000	2001	2002
Ore mined	MBcm	1.8	0.8	0.9	1.2
Waste mined	MBcm	20.4	13.1	20.5	18.9
Ore milled	Mt	3.7	1.8	2.5	3.4
Grade	g/t	4.8	4.5	4.1	4.1
Recovery	%	94.1	88.3	86.8	83.6
Gold produced	ozs	533,675	225,402	291,102	378,317
Cash costs	$/oz	179	272	301	327

It was prudent to benchmark the transition model against a gold mine that had recently made the open-pit to underground transition. The SDGM transitioned to make the mine more economically profitable and hence was suitable as a mine to provide baseline values for the model. After presenting the findings of the study to AGA Continental Africa Region (CAR) senior managers in August 2012, they advised that other mines should be included during the validation process for a more acceptable model. Due to the reason above the gold mining industry norms were also used for comparison. The transition indicators widely used by the mining industry are NPV, stripping ratio, mining cost per ton, UG cost per ore ton, cash cost per ton and margin. In general, a stripping ratio of 4 to 17 is considered as a good indication to consider the UG option, NPV can range from a few billion USD to a few hundred million whereas the margin (gold price to cash cost) of 2 is acceptable since the industry average for 2011 was 1.58 (Wright, 2012). Figure 5-2 shows the gold mining margins from 2001-2011.

Figure 5-2: Gold mining margins from 2001-2011 (Source: Wright, 2012)

5.4 Analysis and interpretation of the results

Table 5-3 shows the comparisons of the mine statistics for the case study mines. Most of the case studies used for OP-UG transition rely on contractors. The treatment methods are CIP Mill suggesting that the deposits may have similar characteristics although each mine is unique. Based on the comparisons a standard for OP-UG transition model could be created.

Table 5-3: Comparisons of mine statistics for the case study mines (Courtesy: AngloGold Ashanti)

MINE		SADIOLA	MORILA	GEITA	CVSA	SUNRISE DAM
MINING RATE	mtpa	16.8	26.3	45.1	17.5	38.2
OPERATOR		Contractor	Contractor	Owner/Contractor	Owner	Contractor
FREEDIG	%	75	40	10	0	0
PROCESS METHOD		Mill	Mill	Mill	Mill	Mill
TREATMENT RATE	mtpa	5.2	3.3	5.3	0.9	3,4
INDICATIVE HEADGRADE	g/t	3	4.7	4.2	8.5	4.7
INDICATIVE MINING COST	US$/t	1.6	1.1	1.1	1.1	1.1
TOTAL COST 2002	US$/Oz	163	74	175	104	177

The normal distribution was a best fit for the results generated for each transition indicator. Hence the results from the results from the OP-UG transition study mines were analysed using normal and cumulative probability distributions to predict the required probabilities for the transition indicators. The number of standard deviations about the mean may be represented by probabilities. If data are normally distributed as in Figure 5-3, then 99.73% of values should fall between ± 3σ while 95.2% of values fall between ± 2σ and 68.26% of the values fall between ± 1σ. Cumulative distributions for parameters, as in the case of the transition indicators, can be used to present and analyse the results. From the graph the probability at 90% and 95% were extrapolated for all the three options that were used to select which of the options best fit the case study deposit. The cumulative distribution graphs for all case study mines are presented in Figure 5-4 to Figure 5-23. The data for the cumulative distributions for all case study mines are shown in Appendix 8.

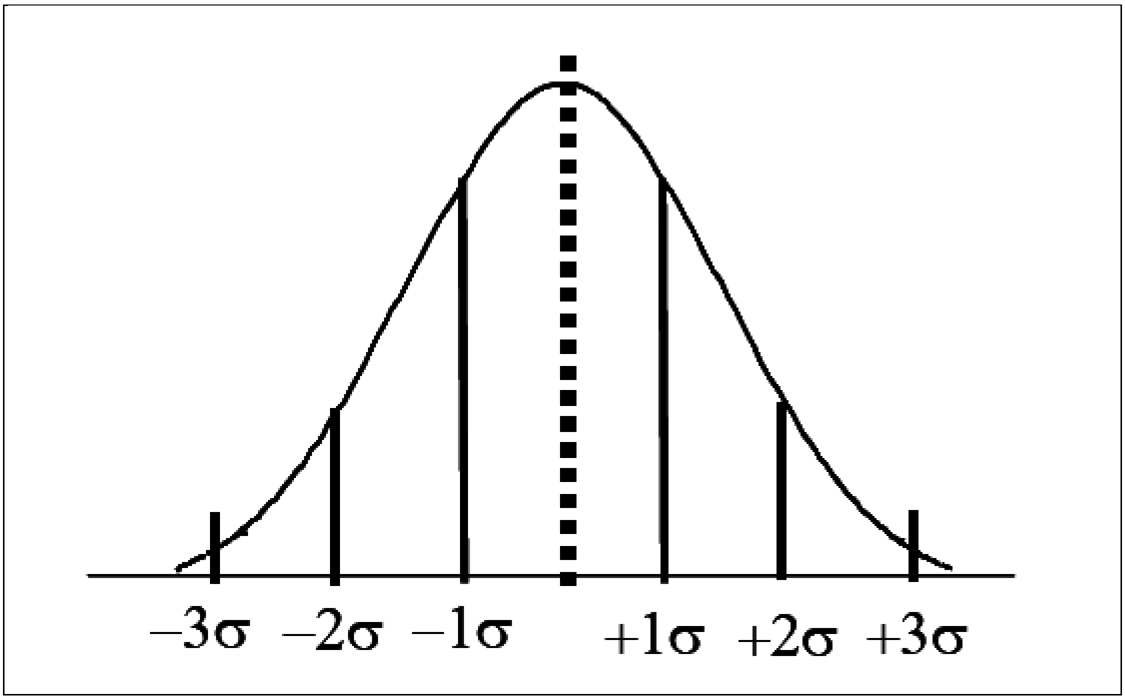

Figure 5-3: Properties of normal distribution

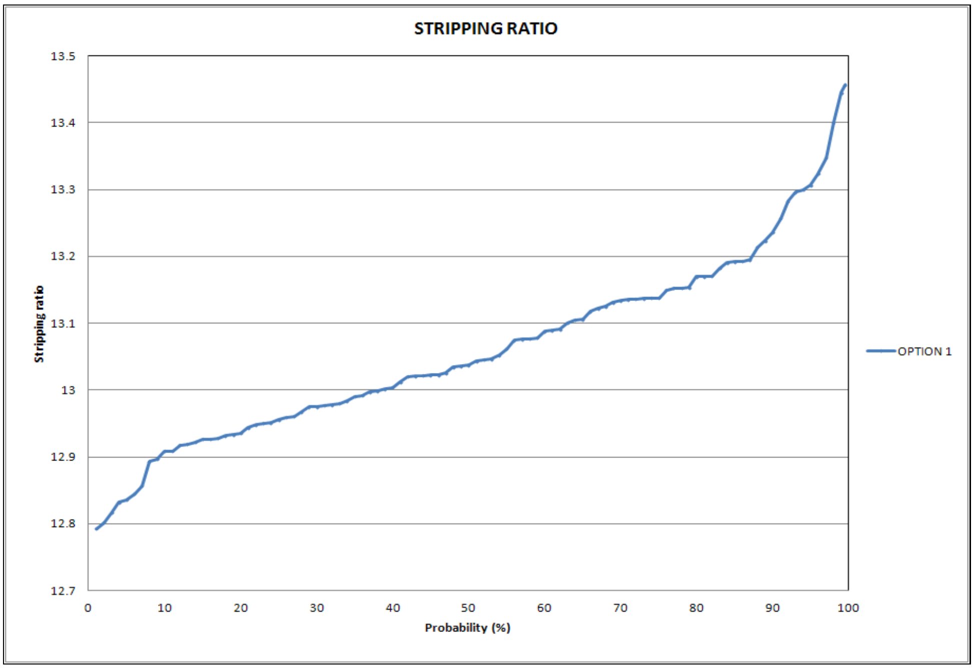

Figure 5-4: Cumulative distribution for CVSA stripping ratio for Option 1

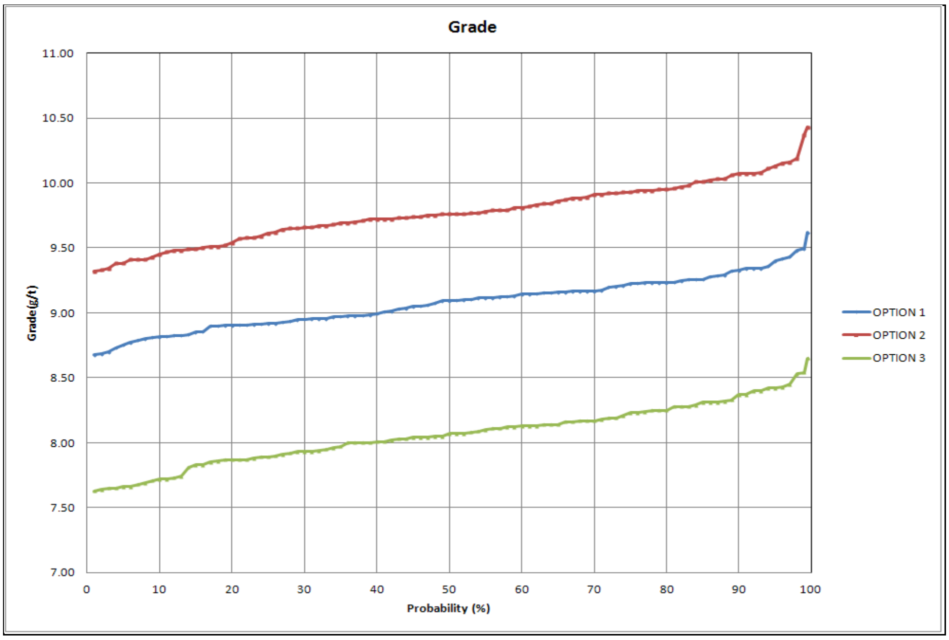

Figure 5-5: Cumulative distribution for CVSA grade for the three Options

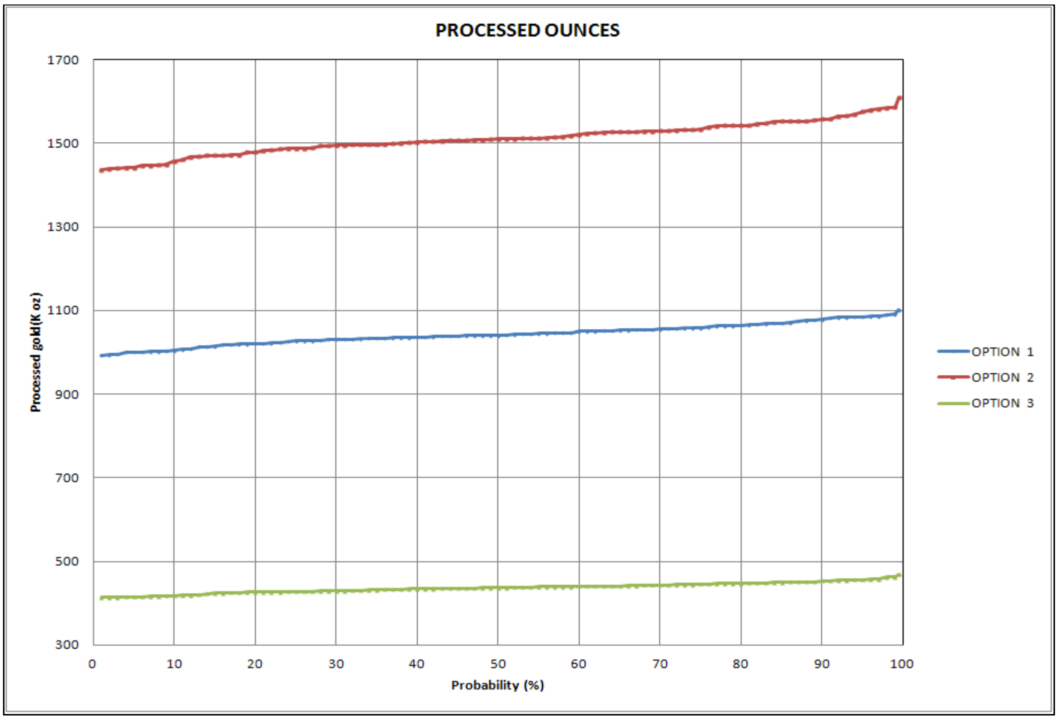

Figure 5-6: Cumulative distribution for CVSA processed ounces for the three Options

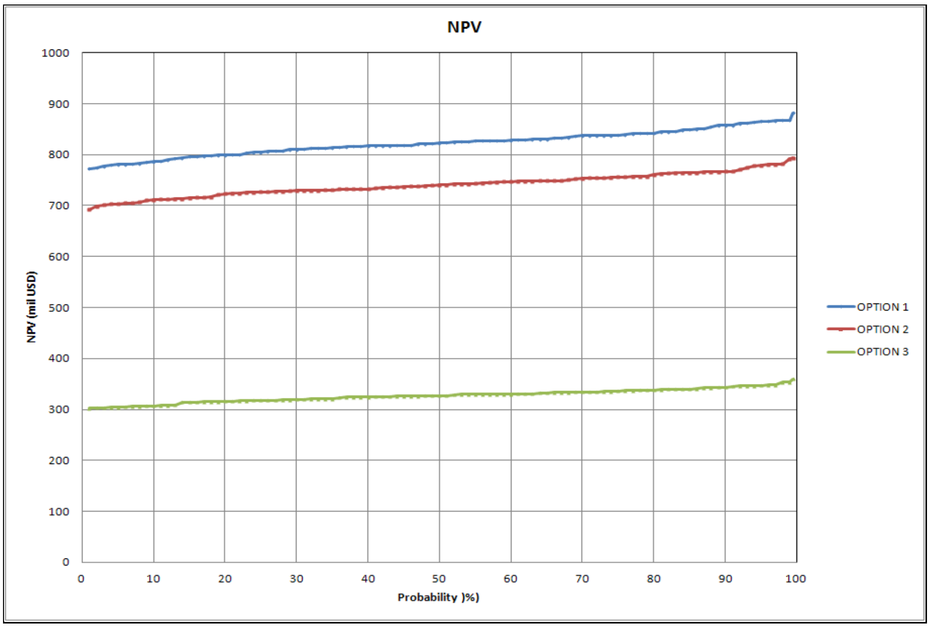

Figure 5-7: Cumulative distribution for CVSA NPV for the three Options

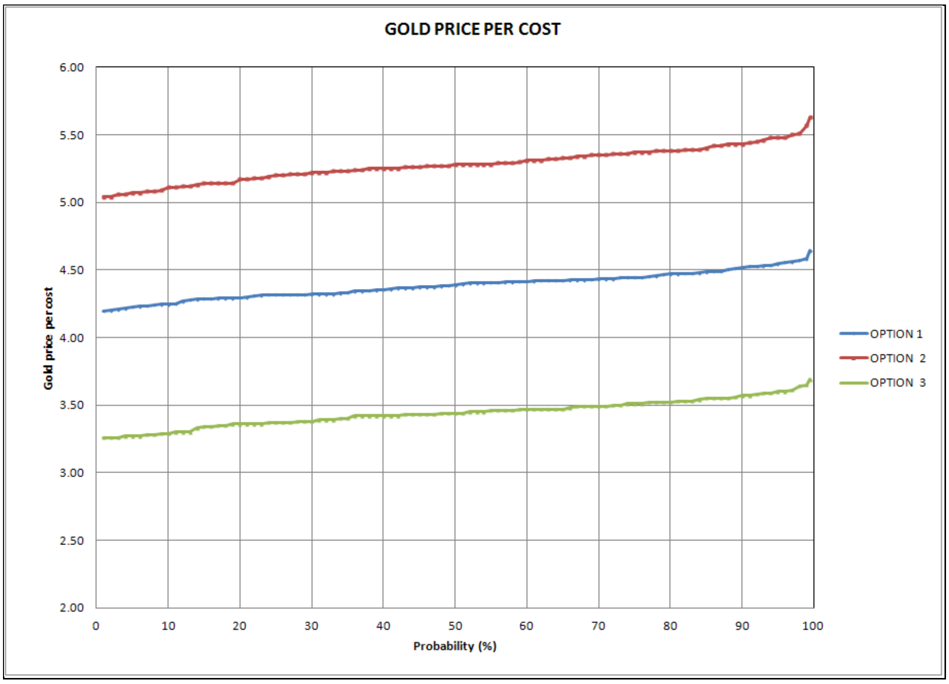

Figure 5-8: Cumulative distribution for CVSA gold price per cost for the three options

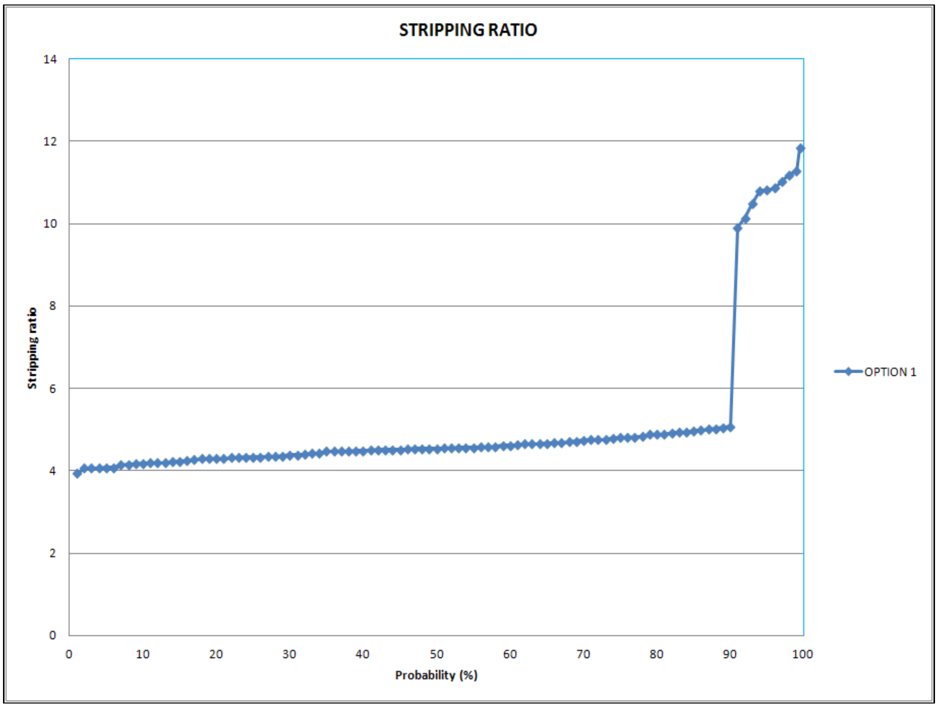

Figure 5-9: Cumulative distribution for Geita stripping ratio for Option 1

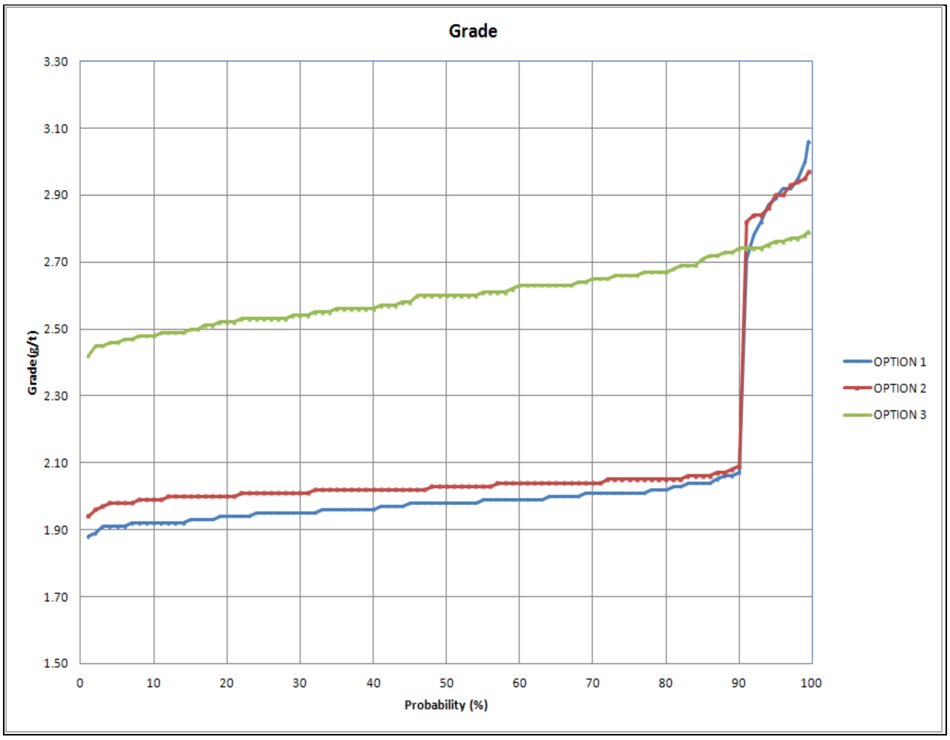

Figure 5-10: Cumulative distribution for Geita grade for Options 1 to 3

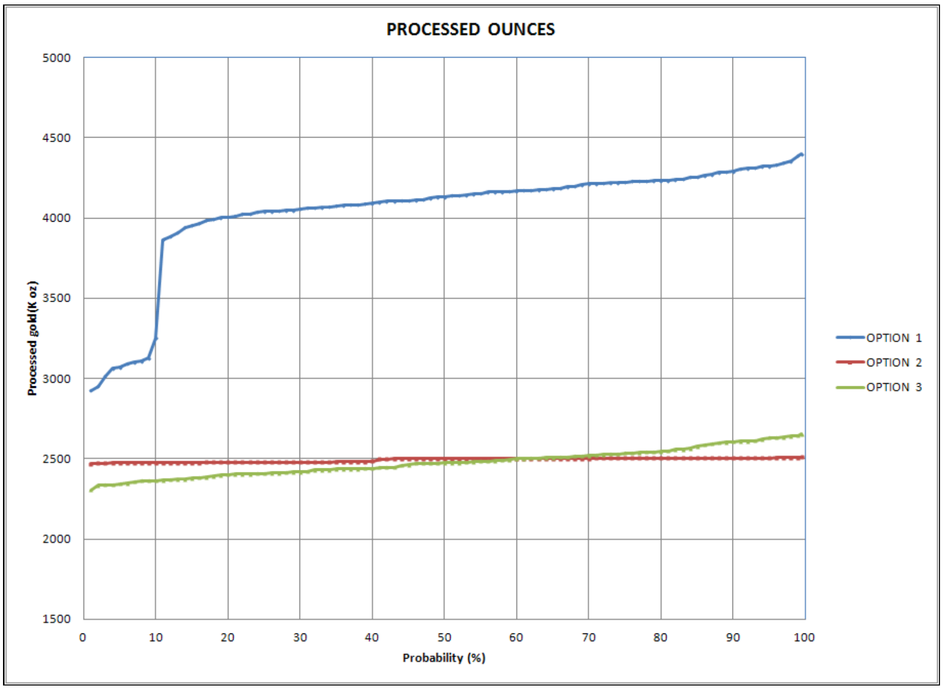

Figure 5-11: Cumulative distribution for Geita processed ounces for Options 1 to 3

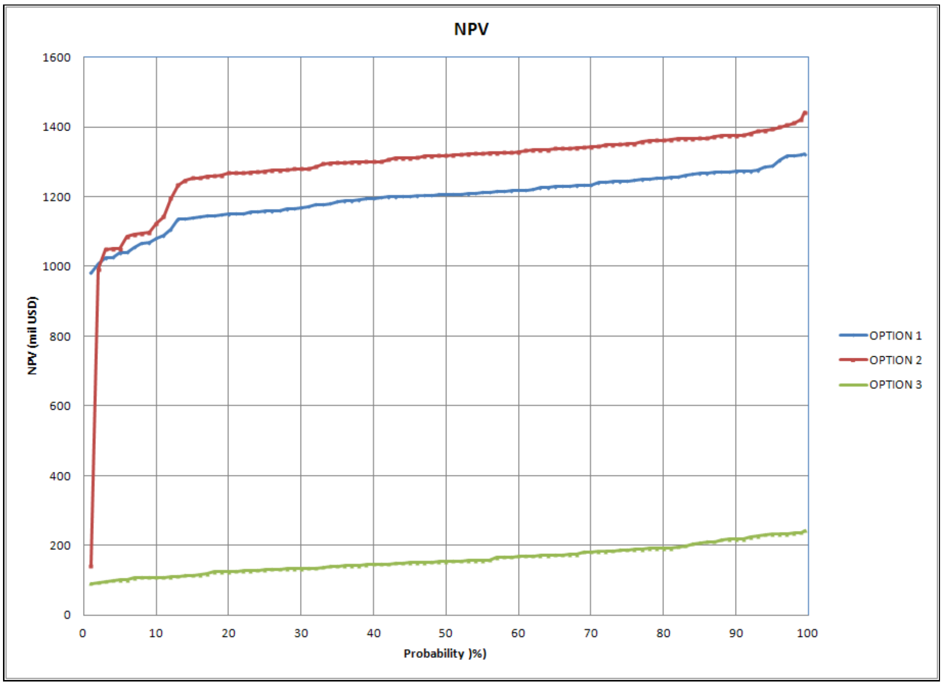

Figure 5-12: Cumulative distribution for Geita NPV for Options 1 to 3

Figure 5-13: Cumulative distribution for Geita Gold price to cost for Options 1 to 3

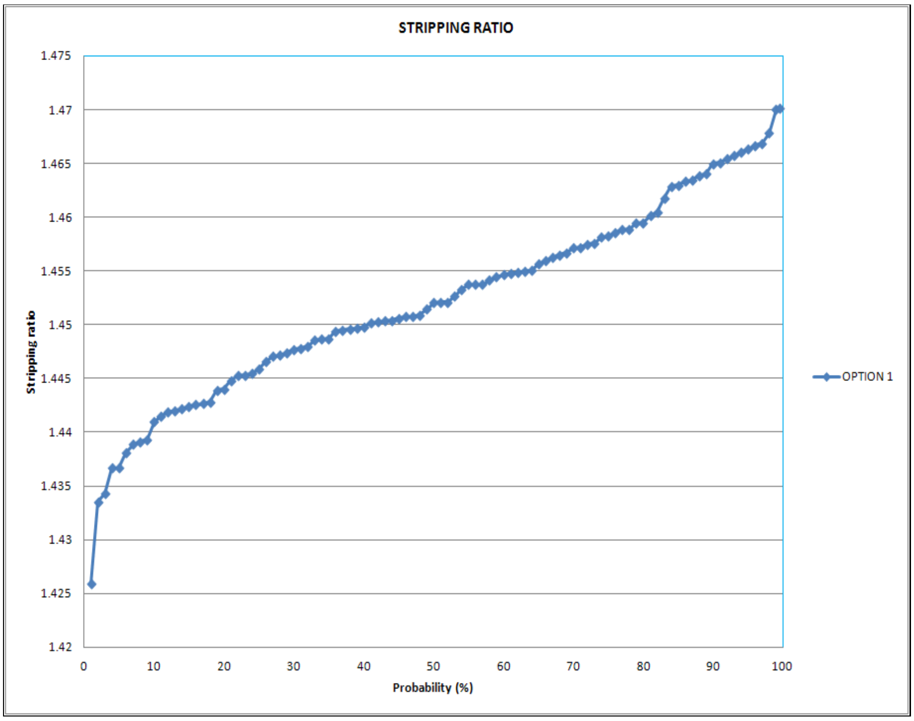

Figure 5-14: Cumulative distribution for Sadiola stripping ratio for Option

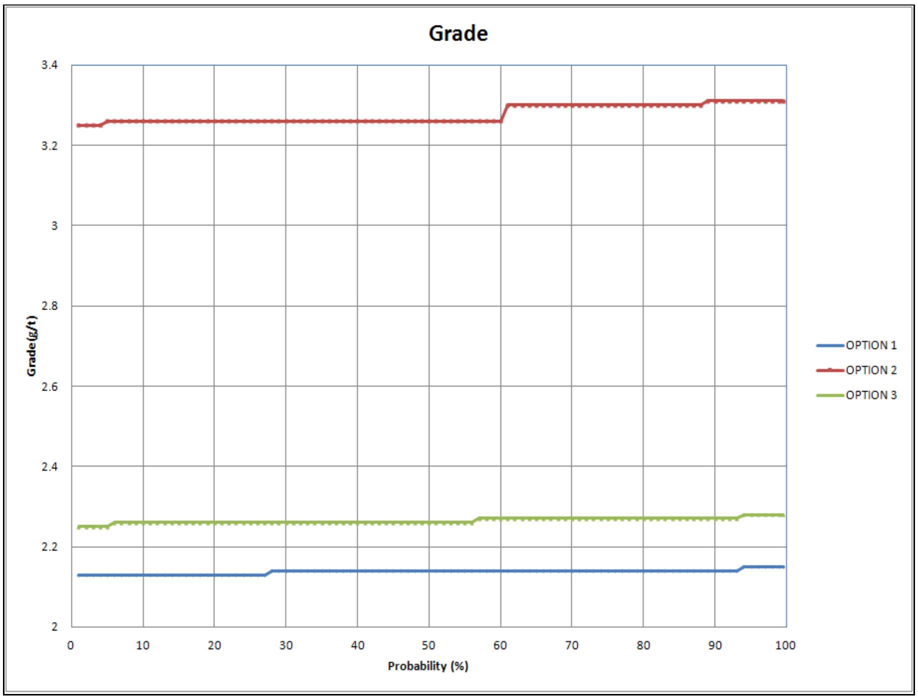

Figure 5-15: Cumulative distribution for Sadiola grade for Options 1 to 3

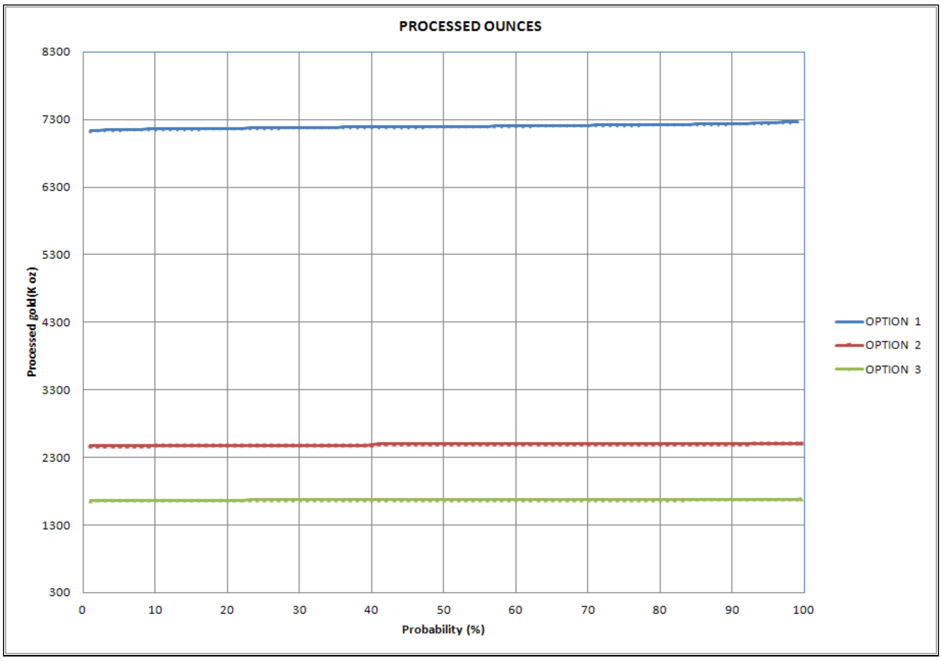

Figure 5-16: Cumulative distribution for Sadiola processed ounces for Options 1 to 3

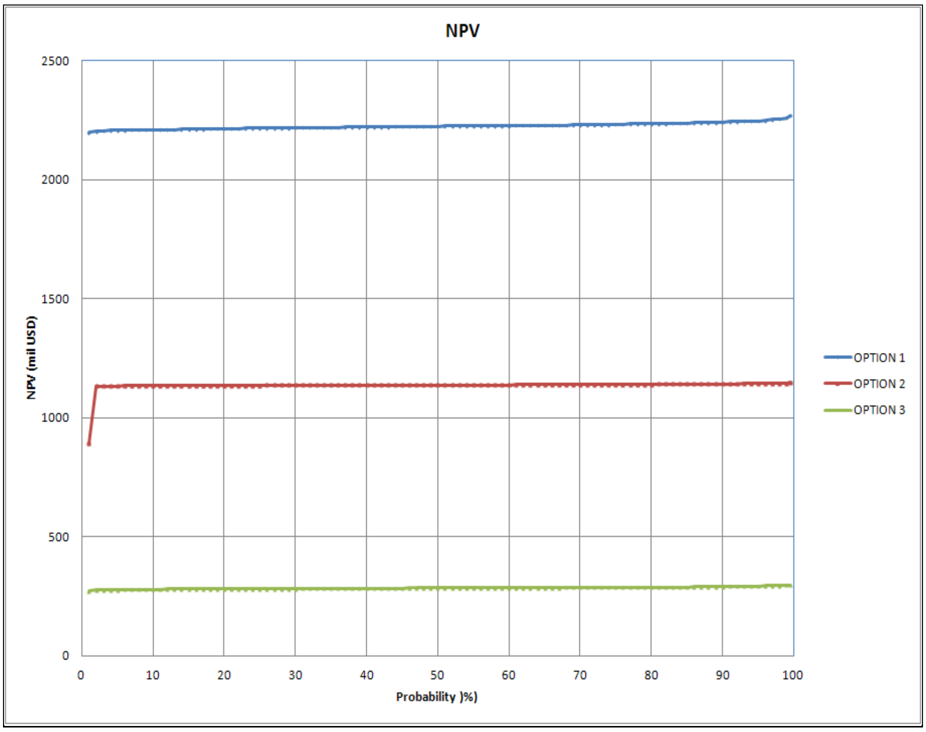

Figure 5-17: Cumulative distribution for Sadiola NPV for Options 1 to 3

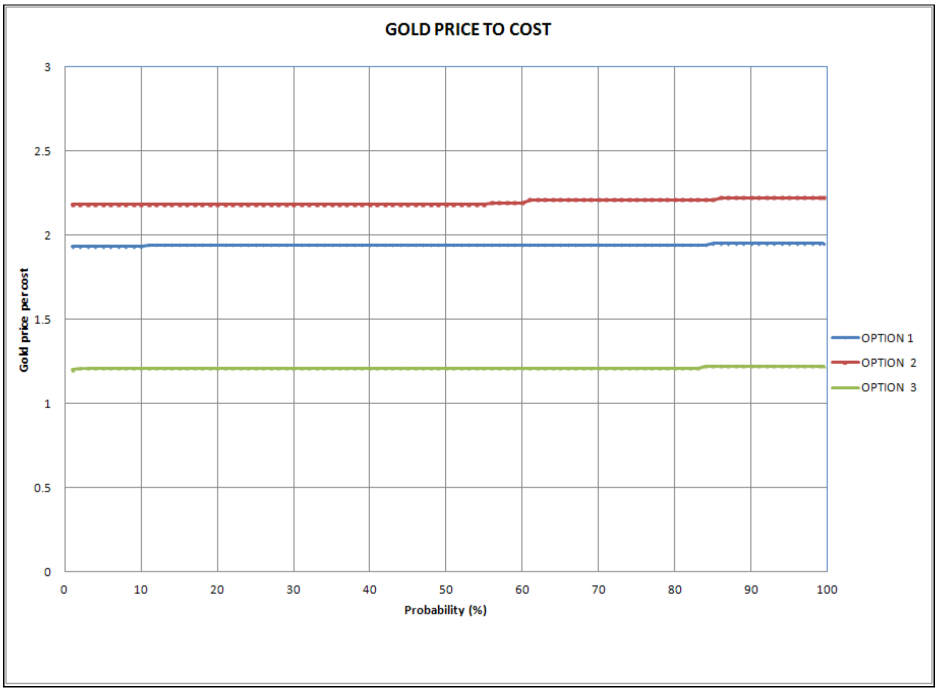

Figure 5-18: Cumulative distribution for Sadiola gold price to cost for Options 1 to 3

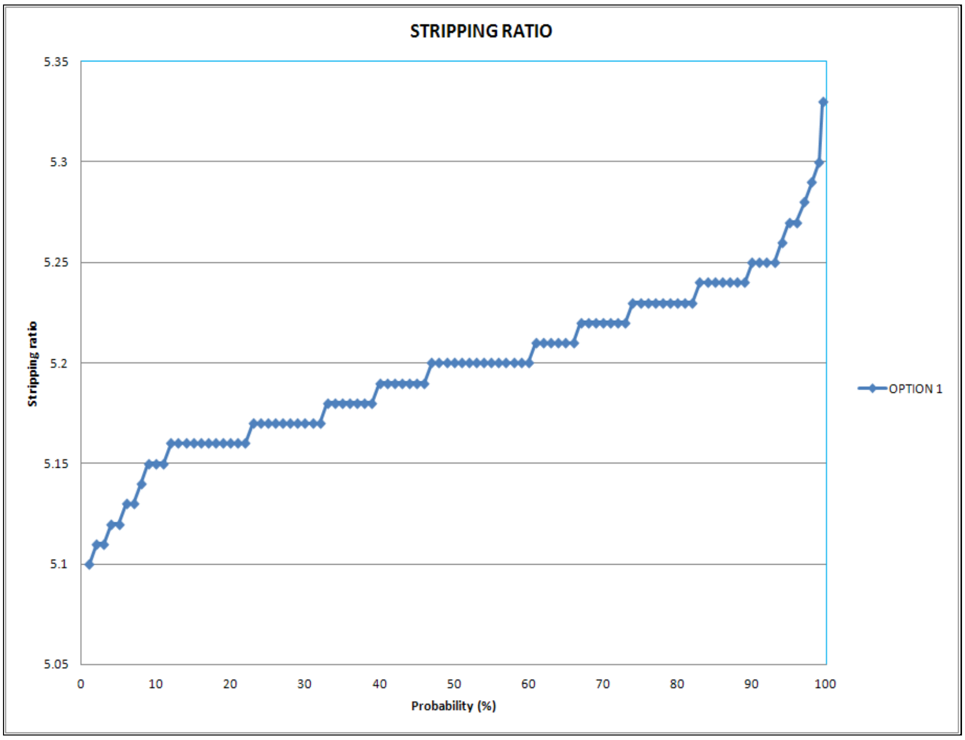

Figure 5-19: Cumulative distribution for Morila stripping ratio for Option 1

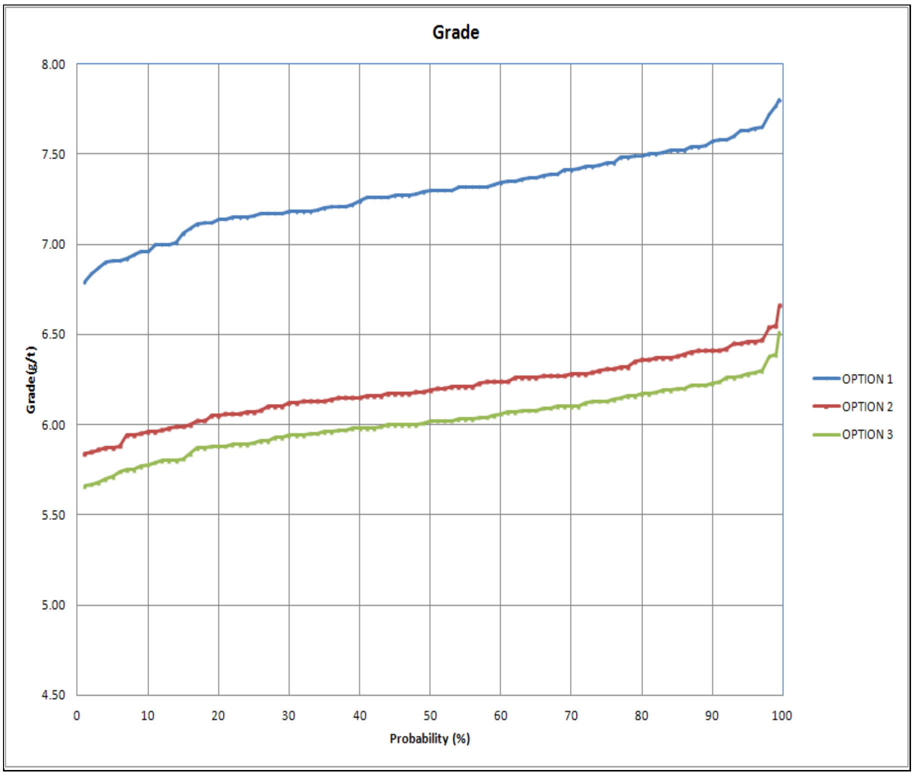

Figure 5-20: Cumulative distribution for Morila grade for Options 1 to 3

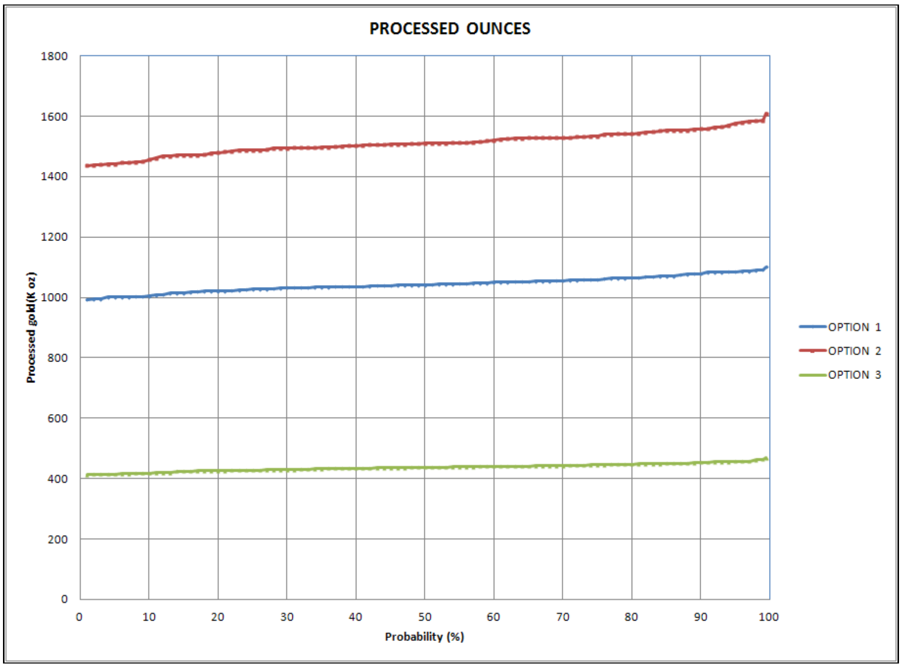

Figure 5-21: Cumulative distribution for Morila processed ounces for Options 1 to 3

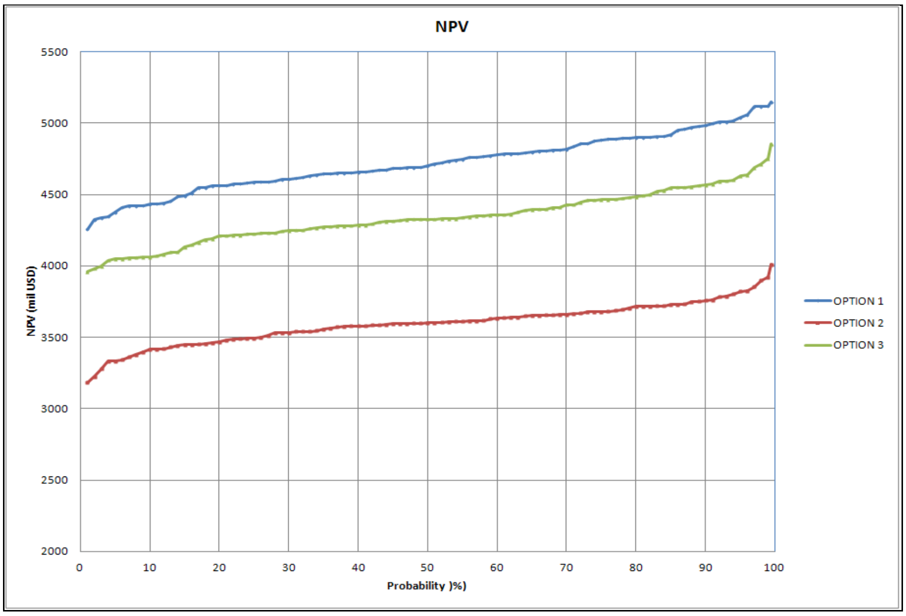

Figure 5-22: Cumulative distribution for Morila NPV for Options 1 to 3

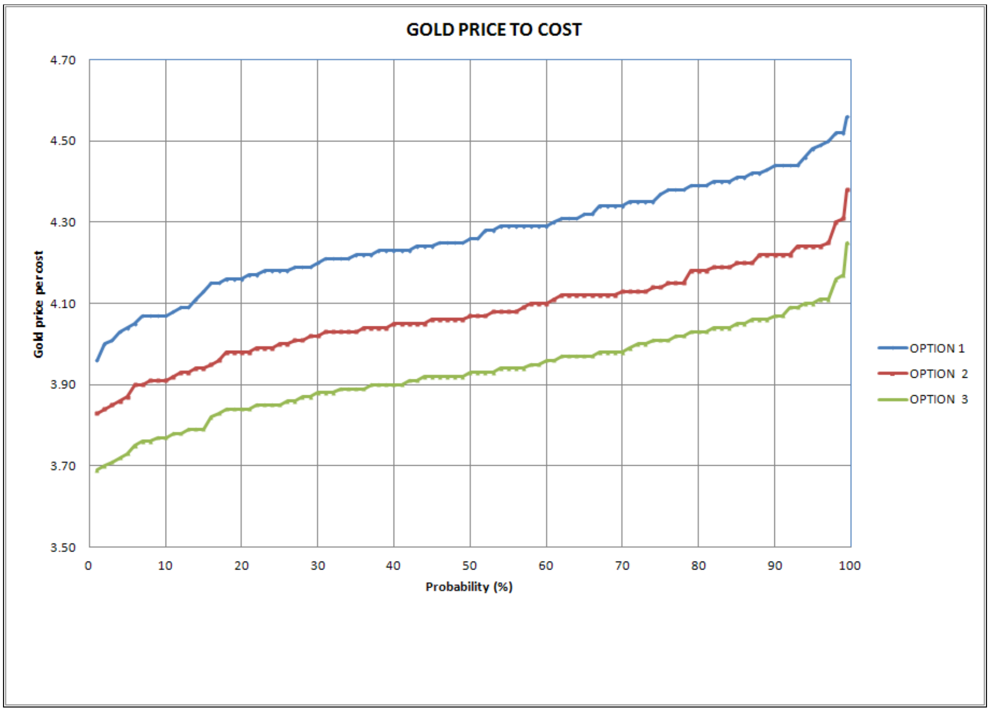

Figure 5-23: Cumulative distribution for Morila gold price to cost for Options 1 to 3

The CVSA case study mine simulated model after optimisation as shown in Figure 5-24 was used to explain how the OP-UG decision should be treated using the results from the simulation to maximise value. The figure shows the pit design done with a gold price of USD 850/oz. The pit designs are shown in Figure 5-24 as well as the pit shells from the optimisation are shown in blue. The pit has been divided into 3 sections for analysis purposes since each section behaved differently namely PIT 1, PIT 2 and PIT 3. From Figure 5-24, PIT 1 can be mine from both OP and UG while Pit 2 and 3 can be mined only from OP for better value. Each line in blue represents each realisation, hence the thickness of the shells shows the variability of the grade in the model.

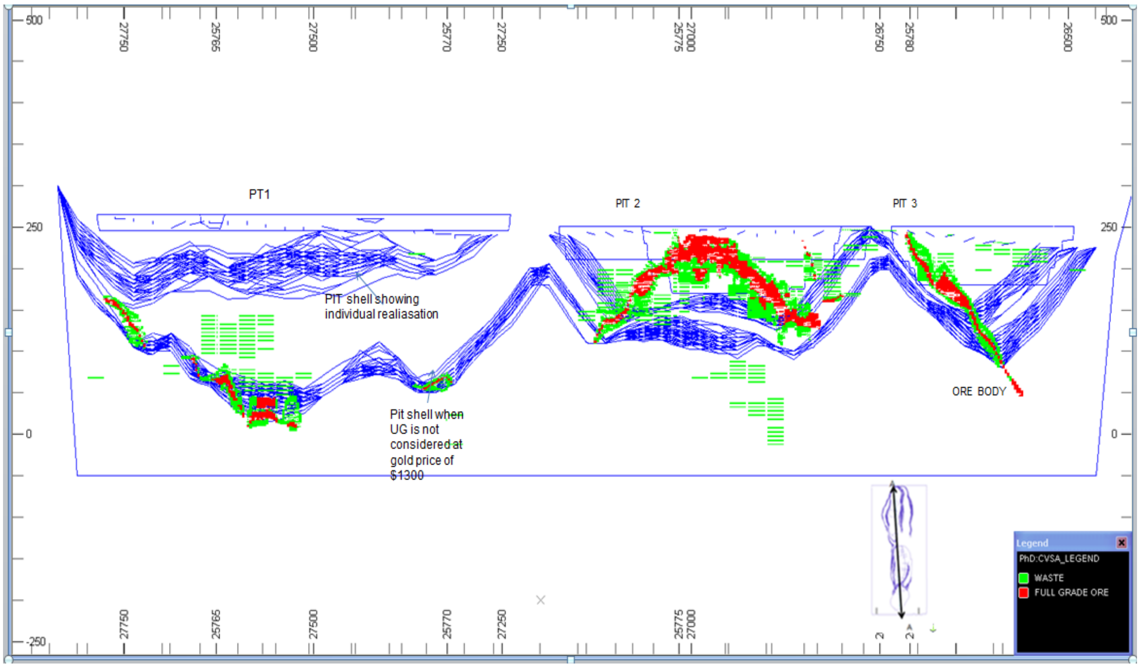

Figure 5-24: Sectional view of CVSA showing pit outlines

The normal distribution curves for 4 different results for Sadiola for Option 2 produced a bi-modal distribution while Geita showed 8 bi-modal distributions for Options 1 and 2. Figure 5-25 shows the Sadiola recovered gold for Option 2 with a bi-modal distribution. Bi-modal distributions occur with some of the case studies because the data used to produce the simulated models were not adequate and sparse, thus forming non-homogeneous distributions. The data were separated to Options 1 and 2 as shown in Figure 5-26 and Figure 5-27. The other results for the bi-modal distributions are shown together with the other histogram results in Appendices 9 to 12.

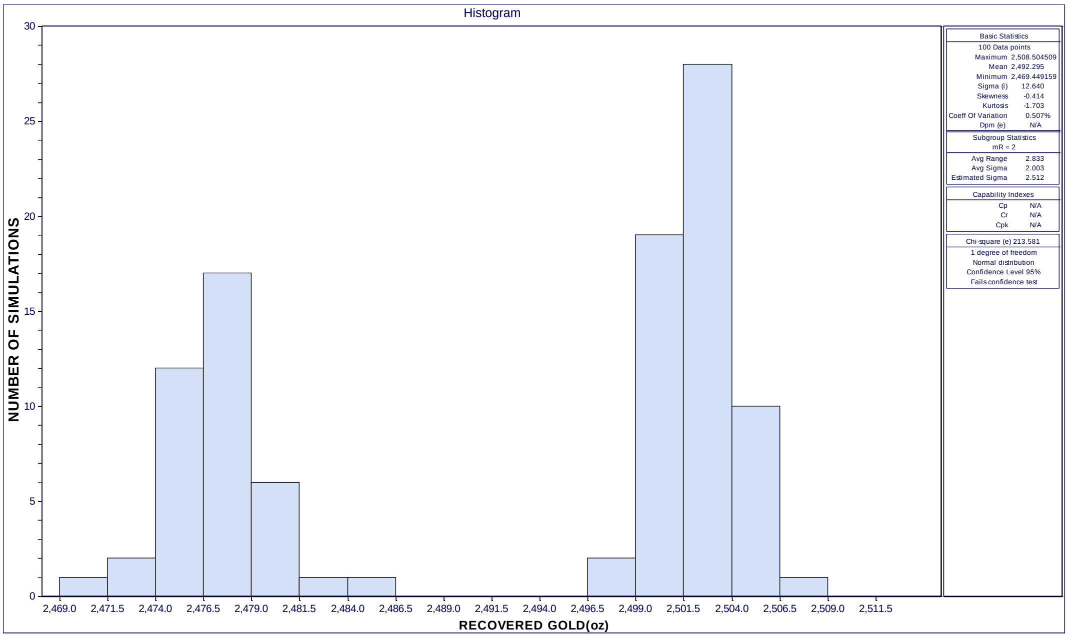

Figure 5-25: Sadiola recovered gold for Option 2 with Bi-modal distribution

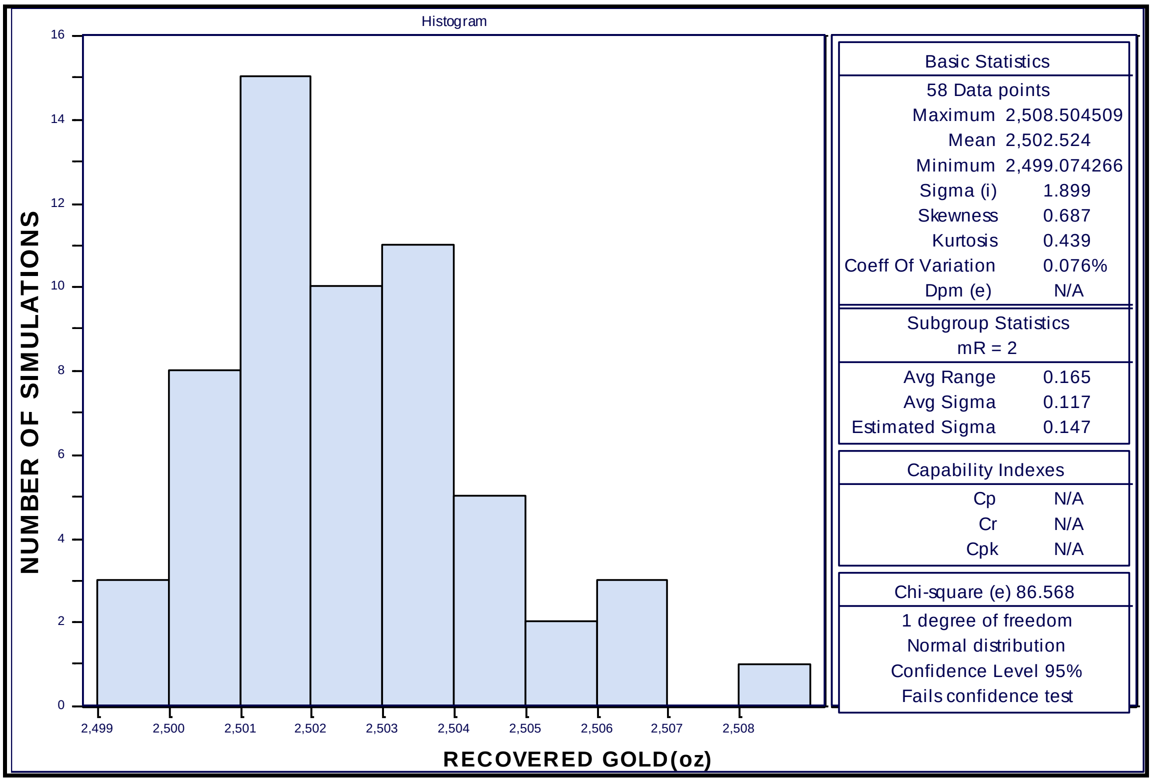

Figure 5-26 : Sadiola recovered gold for Option 2 with Bi-modal Option 1

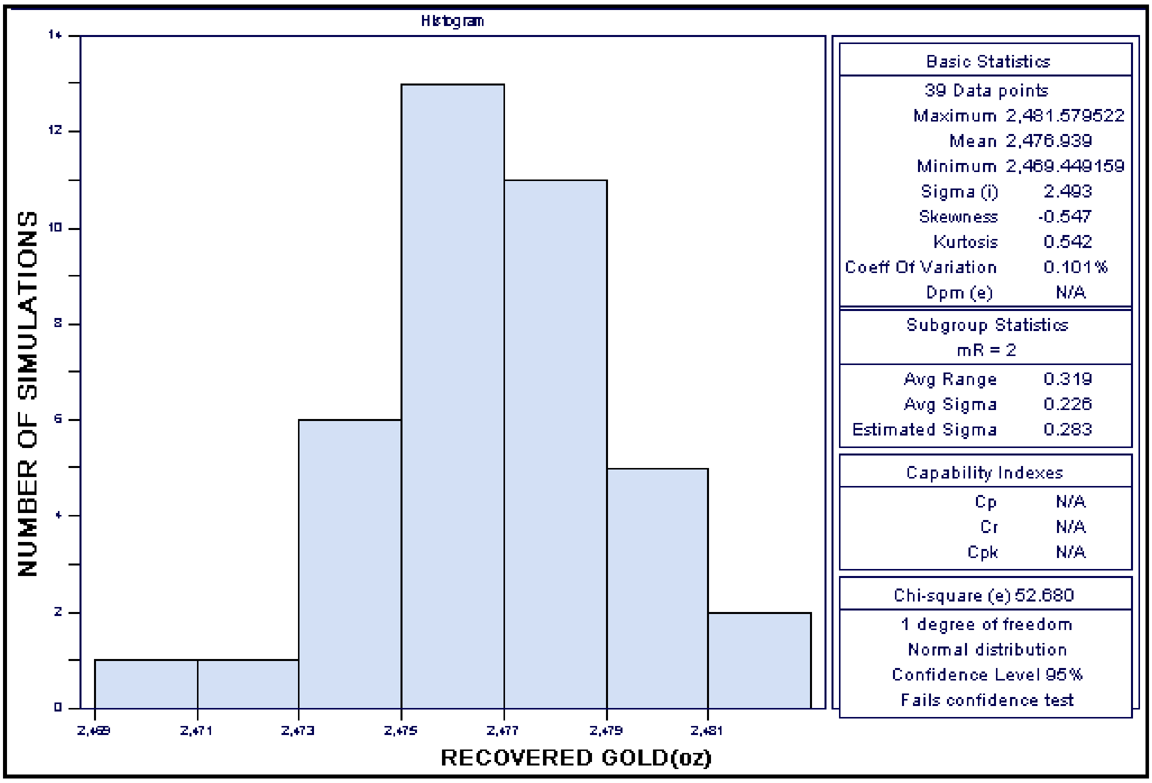

Figure 5-27 : Sadiola recovered gold for Option 2 Bi-modal Option 2

5.5 OP - UG transition model baseline results

The OP-UG transition indicators for SDGM were derived by recognising the following:

- Underground mining operation in SDGM was developed to supplement the open pit not to replace it;
- UG mining operations were to continue production after the end of the open pit operations;
- The UG was planned to start production in 2007;
- Two years were utilised for the transition studies (conceptual, pre-feasibility and feasibility). The transition or lead-time was 3-4 years which was sufficient for the development of the underground mine comprising decline development and underground drilling. The underground transition model in this thesis therefore assumes the minimum time to transition as 3-4 years;
- The initial programme for the pit operations ceased in mid-2008;
- Deep drilling and structural studies led to exploration potential below the Cleo open pit at Sunrise Dam; and
- Underground cut-off grade was 3 g/t with approximately 4-5 million reserve ounces.

The results of the transition indicators for SDGM are included in Table 5-4 and Table 5-5. Stripping ratio has null entries for Option 2 and 3 because these options include underground mining. Table 5-5 compares the summary of the transition indicators with the base model for the various case study mines against SDGM transition indicators. The differences, similarities and patterns between the various transition indicators and processes were compared against each other and with Sunrise Dam Gold Mine. The three options for the four case study mines were as follows:

- Option 1 looks at mining the entire deposit by open pit alone;
- Option 2 considers mining part of the deposit by open pit to the transition point and the rest by underground mining methods; and
- Option 3 considers mining the entire deposit within Option 1 by underground mining methods. The part of the deposit below the Option 1 pit when drilled was considered to be an upside for the underground mining part.

Table 5-4: Key transition indicators for base model

	OPTION	SR	COST (USD/oz)	Processed Gold(k oz)	AVERAGE GRADE(g/t)	NPV(USD mil)	Gold price / Cost
SUNRISE DAM		15.85	362.00	1,426	4.40	688	3.85
	1	1.06	632.11	8878.25	2.22	3053.54	2.06
SADIOLA	2	0.00	467.46	3193.23	3.26	1669.51	2.78
	3	0.00	1100.62	1630.01	2.21	257.05	1.18
	1	14.85	322.77	1059.09	8.67	940.88	4.03
CVSA	2	0.00	283.13	943.13	9.71	456.17	4.59
	3	0.00	323.26	510.71	9.42	401.41	4.02
	1	3.74	436.97	5102.07	4.55	3413.60	2.97
MORILA	2	0.00	385.46	5915.81	5.08	2624.46	3.37
	3	0.00	396.80	5579.56	5.02	3416.08	3.28
	1	24.18	879.90	6223.84	3.73	2376.94	1.48
GEITA	2	0.00	714.25	6569.02	2.88	1307.96	1.82
	3	0.00	715.11	3706.57	3.89	729.94	1.82

Table 5-5: Key transition indicators for simulated models

Mine	OPTION	SR	COST (USD/oz)	Processed Gold(k oz)	AVERAGE GRADE(g/t)	NPV(USD mil)	Gold price / Cost
SUNRISE DAM		15.85	362.00	1,425.54	4.40	688	3.85
	1 (Min)	1.43	665.64	7130.43	2.13	2199.07	1.93
	Ave	1.45	669.94	7199.09	2.14	2224.99	1.94
	Max	1.47	673.79	7275.44	2.15	2268.40	1.95
SADIOLA	2 (Min)	0.00	467.46	2469.45	3.25	892.00	2.18
	Ave	0.00	590.86	2499.48	3.28	1140.74	2.20
	Max	0.00	596.02	3193.23	3.31	1669.51	2.78
	3 (Min)	0.00	1064.20	1661.97	2.25	273.41	1.20
	Ave	0.00	1072.31	1673.05	2.26	284.96	1.21
	Max	0.00	1079.45	1685.79	2.28	297.91	1.22
	1 (Min)	12.79	280.00	993.46	8.68	772.52	4.20
	Ave	13.06	296.56	1043.24	9.07	822.63	4.39
	Max	13.46	309.84	1101.28	9.62	881.79	4.64
	2 (Min)	0.00	230.91	943.13	9.32	456.17	4.59
CVSA	Ave	0.00	246.68	1506.20	9.77	738.26	5.27
	Max	0.00	283.13	1608.35	10.43	793.64	5.63
	3 (Min)	0.00	352.26	413.46	7.63	302.35	3.26
	Ave	0.00	377.99	437.11	8.06	327.24	3.44
	Max	0.00	399.29	468.67	8.65	359.50	3.69
	1 (Min)	5.10	285.36	5788.57	6.79	4257.02	3.96
	Ave	5.20	304.85	6254.89	7.29	4720.69	4.27
	Max	5.33	328.06	6662.64	7.80	5146.20	4.56
	2 (Min)	0.00	296.97	5915.81	5.08	2624.46	3.37
MORILA	Ave	0.00	319.99	7032.51	6.18	3585.98	4.07
	Max	0.07	385.46	7716.60	6.66	4010.42	4.38
	3 (Min)	0.00	305.56	6289.39	5.66	3958.63	3.69
	Ave	0.00	330.93	6695.45	6.02	4337.38	3.93
	Max	0.00	352.02	7245.60	6.51	4853.91	4.25
	1 (Min)	3.94	628.93	2927.39	1.88	981.43	1.60
	Ave	5.15	769.68	4041.42	2.07	1196.84	1.69
	Max	11.85	814.56	4397.36	3.06	1322.50	2.07
	2 (Min)	0.00	714.25	5750.81	1.94	142.03	1.53
GEITA	Ave	0.00	818.55	7920.92	2.12	1287.66	1.59
	Max	0.00	849.04	8529.55	2.97	1441.63	1.82
	3 (Min)	0.00	1002.27	2304.81	2.42	90.79	1.13
	Ave	0.00	1071.32	2477.07	2.60	159.93	1.21
	Max	0.00	1150.04	2652.91	2.79	242.41	1.30

From Table 5-4 and Table 5-5 the following points can be noted:

- Only the underground portion within Option 1 was considered for scheduling with the remaining underground potential below the pit shell as an upside to be explored later and not included for the calculations for Options 2 and 3 for the purpose of this thesis;

- Mineable reserve optimiser (MRO) was used for the underground stopes and the mining methods based on the geometry of the ore bodies and the geotechnical properties;
- Stockpiles were assumed to be available during the transition; and
- Escalations were not applied.

Weights were assigned to the transition indicators as shown in Table 5-6 and Table 5-7. The following points should be noted on how the weightings of the transition indicators were allocated on a scale of 1 to 3. The cost with the lowest value among the three Options (1 to 3) was assigned a weight of 3, because low cost is preferred when mining the deposit whilst the biggest among the 3 is given a weight of 1. For the grade: the grade with the biggest value among the three Options (1 to 3) was assigned a value of 3 and the one with least value a weight of 1. NPV, processed gold and ratio of gold price to cost were assigned weights similar to the grade. The following can therefore be noted:

- Stripping ratio: the smaller the stripping ratio the better it is for the operation to mine using Option 1, but the stripping ratio for Option 2 and 3 to not represent the true value;
- ROM grade: the bigger the value the better it is;
- Cost per ounce: the smaller the value the better for the operation;
- Processed gold: the bigger the value the better it is;
- NPV: the bigger the NPV, the better for the operation to make more profit; and
- Gold price to cost: the bigger the number the better the factor.

After assigning weights to the transition indicators to select the best option out of the three options for the base and the simulated models, Table 5-6 and Table 5-7 were obtained, for the base and the simulated models, respectively. In summary, Option 2 was the preferred option for CVSA while Option 1 was selected for both Morila and the Sadiola deposits based on the highest total weight of transition indicators.

Table 5-6: Summary of weighted transition indicators for base model

Case study	indicator	SR	Processed Gold(k oz)	AVERAGE GRADE(g/t)	NPV(USD mil)	Gold price over cost	Total	Comment
	1	3	3	2	3	2	15	Preferred option
Sadiola	2	2	2	3	2	3	15	
	3	1	1	1	1	1	6	
	1	3	3	2	3	2	15	
CVSA	2	2	2	3	2	3	13	Preferred option
	3	1	1	1	1	1	8	
	1	3	1	1	3	3	14	Preferred option
Morila	2	2	2	3	1	2	11	
	3	1	3	2	2	1	11	
	1	3	2	2	2	3	15	Preferred option
Geita	2	2	3	1	3	2	13	
	3	1	1	3	1	2	9	

Table 5-7: Summary of weighted transition indicators for simulated model

Case study	indicator	SR	Processed Gold(k oz)	AVERAGE GRADE(g/t)	NPV(USD mil)	Gold price over cost	Total	Comment
	1	3	3	1	3	2	15	Preferred option
Sadiola	2	2	2	3	2	3	14	
	3	1	1	2	1	1	7	
	1	3	2	2	3	3	15	
CVSA	2	2	3	3	2	2	15	Preferred option
	3	1	1	1	1	1	6	
	1	3	3	3	3	2	17	Preferred option
Morila	2	2	1	2	1	3	11	
	3	1	2	1	2	1	8	
	1	3	2	1	2	3	14	Preferred option
Geita	2	2	3	2	3	2	14	
	3	1	1	3	1	1	8	

When Sadiola Mine transition indicators were evaluated against the SDGM indicators the following can be noted:

- The stripping ratio of Sadiola is 1.06 compared to 15.85 for SDGM but the cost per ounce is USD 632.11 for Sadiola compared to USD 362 for SDGM thus indicating that one of the criteria for the decision not to transition is being satisfied;
- The average grade is 2.22 g/t compared to 4.40 g/t for SDGM, making the processed ounces for Option 1 equal to 8878.25 koz, about three times that of Option 2 and 3 thus making the transition prohibitive;
- The ratio of gold price to cost per ounce is 2.06 compared to 3.85 for SDGM and slightly above the break-even point. Therefore there is no fundamental difference between the three options; and
- The NPV for Sadiola for Option 1 was 3053.54 USD mil compared to 688 USD mil for SDGM which exceeds the other options. However, it cannot make the transition because it does not satisfy the minimum requirements for the other transition indicators.

When CVSA transition indicators are evaluated against the SDGM indicators the following can be noted:

- The stripping ratio of CVSA is 14.85 compared to 15.85 for SDGM while the cost per ounce for Option 1 is 322.77 USD compared to 362 USD for SDGM within the same range, thus suggesting that one of the criteria for transition is satisfied;
- The ratio of gold price to cost per ounce is 2.06 compared to 3.85 for SDGM, which also satisfies another criterion to transition;
- Although the grade at CVSA is 8.67 g/t compared to 4.40 g/t for SDGM, it has half of the processed ounces of 1059.09 koz compared to 1426 koz for SDGM due to the narrow vein nature of the mineralisation;
- The NPV for CVSA is 940.88 USD mil compared to 688 USD mil for SDGM, suggesting that CVSA could be better than SDGM at transition in terms of NPV;
- From the simulated model results which control the grade variability, this shows slight differences but the overall picture suggests that CVSA should start the process of OP–UG transition with the combined method of both open pit and underground when the transition indicators above are evaluated; and
- Both suggest that it will be best to transition with Option 2.

The evaluation of Morila mine's transition indicators against SDGM's shows the following:

- The stripping ratio is 3.74 compared to 15.85 for SDGM, while the cost per ounce is 436.97 USD compared to 362 USD for SDGM, in the same range suggesting that one of the criteria for the decision not to transition is upheld;
- The average grade is 4.55 g/t compared to 4.40 g/t for SDGM making the processed ounces equal to 5102.07 koz compared to 1426 koz for SDGM;
- The ratio of gold price to cost per ounce is 2.97 compared to 3.85 for SDGM also satisfying another criterion not to transition;
- The NPV is 3413.60 USD mil compared to 688 USD mil for SDGM. Both NPV and gold price to cost ratio at Morila are higher than SDGM suggesting that there is no need to transition but continue mining the pit by means of open pit mining;
- From the results of simulated models there are slight differences but the overall picture suggests that Morila should not transition; and
- Both Table 5-6 and Table 5-7 suggest that it will be best not to transition but continue mining with Option 1.

The following are the similarities and differences between Geita Gold Mine transition indicators and SDGM indicators:

- The stripping ratio is 24.18 compared to 15.85 for SDGM, far more by about a third of that of SDGM while the cost per ounce is 879.90 USD;
- The average grade is 3.73 g/t compared to 4.40 g/t for SDGM but the processed ounces is 6223.84 koz compared to 1426 koz for SDGM;
- The margin is 1.48 compared to 3.85 for SDGM, far less than that of SDGM at transition thus making transition not possible;
- The NPV at Geita is 2376.94 USD mil compared to 688 USD mil for SDGM at transition; and
- Both Table 5-4 and Table 5-5 suggest that it will be best to transition at the current circumstances if the UG grade is doubles that of the current value.

Table 5-8 shows the similarities and differences in values between Sunrise Dam and the case study mines.

Table 5-8: Similarities and differences between Sunrise Dam Gold Mine and the case study mines

	OPTION	SR	COST (USD/oz)	Processed Gold(k oz)	AVERAGE GRADE(g/t)	NPV(USD mil)	Gold price / Cost
SUNRISE DAM		15.85	362.00	1,425.54	4.40	688	3.85
SADIOLA	1	-93%	75%	523%	-50%	344%	-47%
CVSA	1	-6%	-11%	-26%	97%	37%	5%
MORILA	1	-76%	21%	258%	3%	396%	-23%
GEITA	1	53%	143%	337%	-15%	246%	-62%

The various mines in this study were compared to SDGM baseline values in order to select the transition decision. This has been guided by taking into consideration the uncertain nature of the parameters used for optimisation and grade variability in geological models. Table 5-9 and Table 5-10 show a summary of the probabilities of achieving 95% and 90% using cumulative frequency, respectively. The trend from the results shows that with the exception of NPV, all of the indicators favour Option 2 for CVSA deposit and Option 1 for Geita deposit. Option 1 is the preferred option since all the indicators favour it with the exception of processed ounces. Option 1 will be the preferred option for Sadiola Mine. Table 5-4 and Table 5-5 are comparable to the probability tables for achieving 95% and 90% as shown in Table 5-9 and Table 5-10 respectively, which were used to create the transition model flowchart.

Table 5-9: Transition indicator values at 95% cumulative probability

PROBABILITY	INDICATOR	OPTIONS	MINE			
			MORILA	SADIOLA	GEITA	CVSA
95%						
	Stripping Ratio	1	5.27	1.47	10.82	13.31
	Average grade (g/t)	1	7.63	2.15	2.89	9.40
		2	6.46	3.31	2.90	10.13
		3	6.28	2.28	2.76	8.42
	Processed Gold (k oz)	1	1085	7250	4321	1085
		2	1576	2505	2505	1576
		3	457	1681	2630	457
	NPV (USD mil)	1	5040	2246	1288	865
		2	3823	1143	1394	779
		3	4629	293	232	347
	Gold price to cost	1	4.48	1.95	1.99	4.54
		2	4.24	2.22	1.74	5.48
		3	4.10	1.22	1.29	3.60

Table 5-10: Transition indicator values at 90% cumulative probability

PROBABILITY	INDICATOR	OPTIONS	MINE			
			MORILA	SADIOLA	GEITA	CVSA
90%						
	Stripping Ratio	1	5.25	1.46	5.08	13.24
	Average grade (g/t)	1	7.57	2.14	2.07	9.33
		2	6.41	3.31	2.09	10.07
		3	6.23	2.27	2.74	8.37
	Processed Gold (k oz)	1	1078	7238	4290	1078
		2	1558	2504	2504	1558
		3	454	1679	2607	454
	NPV (USD mil)	1	4984	2242	1272	857
		2	3759	1142	1375	767
		3	4565	291	218	343
	Gold price to cost	1	4.44	1.95	1.72	4.52
		2	4.22	2.22	1.62	5.43
		3	4.07	1.22	1.28	3.57

Table 5-11 shows the sensitivities of the transition indicators to gold price. Sensitivities of the Geita deposit gold price indicate that an increase in gold price from $1300 /oz to

$2000/oz will not make much difference unless further drilling is done to improve the grade in the resource below the $1300/oz shell. Sensitivities on the transition timing can be determined by considering the projected cost and the gold price at the time.

Gold price sensitivities simulations were run for USD 700 /oz, USD 850 /oz, USD 950 /oz, USD 1100 /oz, USD 1500 /oz and USD 2000 /oz to see the effect of gold price on the transition indicators. Sensitivities for gold price were done for Option 1 (mining from open pit). Table 5-11 shows the summary of the gold price sensitivities for the case study mines for open pit when underground is not considered while The closer the figures in Table 5-11, the more viable the option to mine the deposit by open pit hence this delays the transition. Analysis of the results of sensitivities suggests that most of the deposits are not sensitive to the stripping ratio, with the exception of CVSA, despite the fact that the stripping ratio is high for Geita and low for Morila and Sadiola mines. The *in-situ* grades for each deposit are shown in Table 5-11.

Table 5-11: Sensitivities of the transition indicators to gold price for Option 1

Mine	Gold price(USD)	SR	Processed Gold(k oz)	AVERAGE GRADE(g/t)	NPV(USD mil)	Gold price / Cost
	700	1.05	2992	3.13	686	1.56
	850	1.44	3945	2.86	1251	1.70
	950	1.67	4550	2.73	3179	2.45
SADIOLA	1100	2.39	5727	2.59	2699	1.89
	1300	2.96	7210	2.46	4378	2.06
	1500	3.33	9241	2.29	6804	2.17
	2000	3.34	12921	1.96	14017	2.48
	700	17.47	935	11.55	377	2.73
	850	15.51	999	9.99	514	2.99
	950	14.94	1024	9.44	607	3.20
CVSA	1100	14.84	1043	9.03	749	3.55
	1300	14.85	1059	8.67	941	4.03
	1500	14.76	1069	8.39	1134	4.50
	2000	14.67	1081	7.99	1622	5.73
	700	3.88	4587	6.47	1622	2.25
	850	3.61	5082	5.61	2306	2.42
	950	3.64	5523	5.05	2808	2.43
MORILA	1100	3.43	5821	4.60	4647	3.08
	1300	3.32	6079	4.24	4676	2.86
	1500	3.40	6363	3.91	5815	3.03
	2000	3.86	6916	3.38	8835	3.36
	700	27.61	1912	5.15	11	1.01
	850	26.76	1912	4.60	323	1.12
	950	26.17	4829	4.35	685	1.20
GEITA	1100	25.06	5534	4.04	1347	1.32
	1300	24.18	6224	3.73	2377	1.48
	1500	23.34	6654	3.50	3520	1.63
	2000	21.67	7317	3.08	6658	2.00

Table 5-12 shows that with the exception of CVSA mine, the other case study mines are not ready to transition under the current circumstances. The information in Table 5-12 together with the Qualitative indicators discussed earlier on, were used to construct the transition model flowchart illustrated by Figure 5-28.

Table 5-12: OP-UG transition indicators in relation to baseline values

MINE	SUNRISE DAM	MORILA	SADIOLA	GEITA	CVSA
Stripping Ratio	15.8	> 3.0	> 3.0	> 11.0	> 13.0
Average grade (g/t)	4.4	> 5.2	> 4.0	> 8.0	> 9.0
Processed Ounces (k oz)	1425	> 1000	> 4000	> 3500	> 1000
NPV (USD mil)	688	> 3000	> 1300	> 1200	> 700
Gold price to cost	3.9	> 2.0	> 2.0	> 2.0	> 2.0

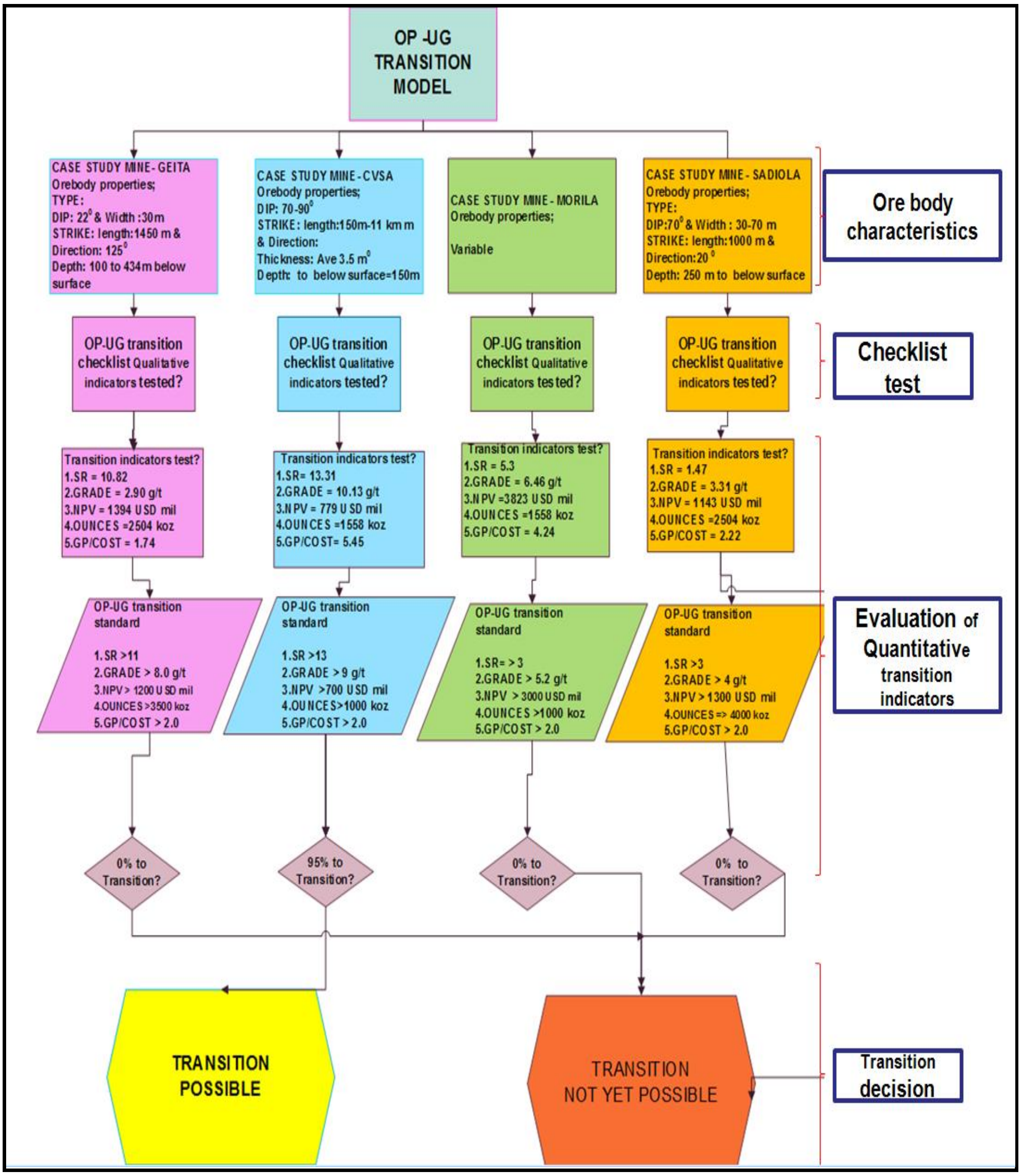

Figure 5-28: OP – UG transition model flowchart

For a mine with similar ore body characteristics to Sadiola to transition, the stripping ratio must be greater than 3, grade must be greater than or equal to 4 g/t and the gold price to cost ratio must be above 2.0 for a probability of 95%. For a mine with similar characteristics to Morila to transition, the stripping ratio must be greater than 3.0, grade must be greater than or equal to 5.2 g/t and the gold price to cost ratio must greater than 2.0 to achieve the probability of 95%. Similarly, for mines with similar ore body characteristics to those of CVSA to transition, the stripping ratio must be greater than 13 and the grade must be greater than 9.0 g/t and the gold price to cost ratio must be at least 2.0 to achieve a probability of 95%. For a mine with similar ore body characteristics to Geita to transition, the stripping ratio must be greater than 11, grade must be greater than or equal to 8.0 g/t and the gold price to cost ratio must be at least 2.0.

5.6 Chapter summary

This chapter has demonstrated how the conceptual model proposed in Chapter 3 was developed into the transition model by using a gold mine that had recently made the open-pit to underground transition as a baseline. The mine used for this purpose was Sunrise Dam Gold Mine. The challenge now is for mine planners in the mining industry to start using simulated geological models to incorporate uncertainty into pit designs to better quantify geological risk, as demonstrated by various authors and supported in this thesis. The next chapter will conclude and recommend on the OP-UG transition model.

6.0 CONCLUSIONS AND RECOMMENDATIONS

6.1 Conclusions

The preliminary findings of this research study were presented to AngloGold Ashanti Continental Africa Region (CAR) senior managers in August 2012 and the feedback obtained was used to further refine the modelling work done. The transition indicators for the four case study deposits indicate that at the probability of 95%, the only mine to transition from OP-UG is the CVSA mine under the prevailing circumstances. The CVSA deposit has all the transition indicators favouring Option 2 than the other options except NPV. The transition indicators in Geita at a probability of 95% favour Option 1. Sadiola deposit at a probability of 95% has values for processed ounces and NPV in favour of Option 1 than Options 2 and 3. Morila deposit at a probability of 95% favours Option 1 for all the transition indicators with the exception of processed ounces.

Some of the observations from the research study include the following:

- Drilling ore bodies to the required level of confidence is key for OP-UG transition timing; there is a need to do adequate pre-drilling to define the geology and for exact location of geological structures;
- The mine evaluation geologist should consider building simulation models for open pit portions of OP-UG transition and a separate Kriged model for the underground portion of the OP-UG transition;
- Underground cost should be determined to the required level of accuracy in order for the OP-UG transition decision to be evaluated because the pit to be mined before transition depends on the underground cost;
- Simulated model aid in quantifying the grade variability in the model;
- It is always better to mine the pit shell when the underground option is considered as a pushback first, followed by the open pit shell with less risk when using simulated models for the OP-UG transition;
- The relationship between grade and cost per ounce is not linear; and
- The transition indicators should be used collectively during OP-UG transition decision-making and not in isolation.

6.2 Research contribution and limitations

The research contributions are as follows:

- The OP – UG transition decision should be treated as a decision to determine the transition point (H_{tp}) rather than transition depth (H_{td});
- A model to guide OP - UG transition decision for gold mines was conceptualised and then developed using actual mine data. Gold mines can use the model and assume 3-4 years as the minimum time to transition provided all transition indicators are met;
- The annual LoM reviews of open pit mines should include assessing the OP-UG interface; and
- The LoM reviews should consider the use of simulated models for generating OP-UG transition indicators to reduce the effect of geological uncertainties in their life of mine plans.

This research assumed that:

- The model is limited to gold mines only. However, the concept that was developed can be applied to other commodities to derive appropriate models; and
- Separate geological Kriged block models could not be obtained from the mines hence the same block models were used for both open pit and underground. However, reconciliation between the SMU and the kriged models were done to determine the difference between them.

6.3 Recommendations for future research

Following on from discussions arising from the research findings, the possible future areas of research should extend the concept of transition indicators to deposits other than gold, preferably with different software, to produce models for other minerals for the open pit to underground transition decision.

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APPENDICES

Appendix 1: Fields in geological block models

Geita

CLASS
0 = Unclassified
2 = Indicated
3 = Inferred

MZONE
0 = Waste
1 = Nyankanga Main
2 = Nyankanga West
3 = Nyankanga Extension (into Lone Cone North)
4 = Dyke

OXIDE
0 = Sulphide
1 = Transition
2 = Oxide

RTYPE
1 = BIF (>80%)
2 = Felsic (>80%)
12 = Mixed BIF & Felsics
3 = Quartz or Feldspar Porphyry Dyke
4 = Dioritic Intrusive
5 = Laterite or Regolith
BIFS_IND = BIFS Proportion
BIFC_IND = BIFC Proportion
FELSPROP = Felsic Proportion

AU = Grade at 0.0 g/t COG
DENSITY = Density

KZONE
1 = ZONE 1 ORE
2 = ZONE 2 ORE
3 = LONE CONE SIDE 3 ORE
4 = BLOCK 1 ORE
5 = NY WEST ORE
6 = DYKE
7 = ZONE 1 WASTE
8 = MAIN ZONE WASTE
9 = LONE CONE SIDE + BLOCK1 WASTE

Sadiola

BLOCK MODEL GEOMETRY

FIELD NAME	DESCRIPTION
IJK	Block index
XC	X centroid of block
YC	Y centroid of block
ZC	Z centroid of block
XINC	Block size in X direction
YINC	Block size in Y direction
ZINC	Block size in Z direction

XMORIG	Block model origin (minimum X)
YMORIG	Block model origin (minimum Y)
ZMORIG	Block model origin (minimum Z)
NX	Number of cells in X direction
NY	Number of cells in Y direction
NZ	Number of cells in Z direction

KZONE
Estimation domains

KZONE	CODE
Waste	10
FW	100
HW	200
NE trend	300
SZ North	410
SZ South – low	421
SZ South – high	422

AU
Kriged grade field
PXXX
SMU proportion above cut-off grade where XXX is the cut-off grade.
For example P050 is the proportion above a cut-off grade of 0.5 g/t.
GXXX
SMU grade above cut-off where XXX is the cut-off grade.
For example G050 is the grade above a cut-off of 0.5 g/t.
CLASS
Classification field

CLASS	CODE
Measured	1
Indicated	2
Inferred	3
Blue Sky Tangible	4
Blue Sky Intangible	5

ROCKTYPE
Material type field

ROCKTYPE	CODE
Laterite & Clay	1
Oxide Saprolite	2
Siliceous Saprolite	3
Sulphidic Saprolite	4
Hard Sulphide	5
Blast Oxide	6
Blast Sulphide	7

LITH
Lithology field

LITH	CODE
Dolerite	1
Meta-greywacke	2
Meta-limestone	3

ROCKCODE Combination of KZONE, LITH & ROCKTYPE fields

Rock Group	Rock Name					Rock Code
SOFTOXID	DIWSTSOX	Diorite	Waste	Soft	Oxide	2100
SOFTOXID	DIHGSOX	Diorite	High Grade	Soft	Oxide	2112

SOFTOXID	DILGSOX	Diorite	Low grade	Soft	Oxide	2113
SOFTSULF	DIWSTSSU	Diorite	Waste	Soft	Sulphide	2200
SOFTSULF	DIHGSSUL	Diorite	High Grade	Soft	Sulphide	2212
SOFTSULF	DILGSSUL	Diorite	Low grade	Soft	Sulphide	2213
HARDOXID	DIWSTHOX	Diorite	Waste	Hard	Oxide	2300
HARDOXID	DIHGHOX	Diorite	High Grade	Hard	Oxide	2312
HARDOXID	DILGHOX	Diorite	Low grade	Hard	Oxide	2313
HARDSULF	DIWSTHSU	Diorite	Waste	Hard	Sulphide	2400
HARDSULF	DIHGHSUL	Diorite	High Grade	Hard	Sulphide	2412
HARDSULF	DILGHSUL	Diorite	Low grade	Hard	Sulphide	2413
SOFTOXID	CMWSTSOX	Marble	Waste	Soft	Oxide	3100
SOFTOXID	CMHGSOX	Marble	High Grade	Soft	Oxide	3112
SOFTOXID	CMLGSOX	Marble	Low grade	Soft	Oxide	3113
SOFTSULF	CMWSTSSU	Marble	Waste	Soft	Sulphide	3200
SOFTSULF	CMHGSSUL	Marble	High Grade	Soft	Sulphide	3212
SOFTSULF	CMLGSSUL	Marble	Low grade	Soft	Sulphide	3213
HARDOXID	CMWSTHOX	Marble	Waste	Hard	Oxide	3300
HARDOXID	CMHGHOX	Marble	High Grade	Hard	Oxide	3312
HARDOXID	CMLGHOX	Marble	Low grade	Hard	Oxide	3313
HARDSULF	CMWSTHSU	Marble	Waste	Hard	Sulphide	3400
HARDSULF	CMHGHSU	Marble	High Grade	Hard	Sulphide	3412
HARDSULF	CMLGHSU	Marble	Low grade	Hard	Sulphide	3413
SOFTOXID	GWASTSOX	Greywacke	Waste	Soft	Oxide	4100
SOFTOXID	GWKHGSOX	Greywacke	High Grade	Soft	Oxide	4112
SOFTOXID	GWKLGSOX	Greywacke	Low grade	Soft	Oxide	4113
SOFTSULF	GWASTSSU	Greywacke	Waste	Soft	Sulphide	4200
SOFTSULF	GWKHGSSU	Greywacke	High Grade	Soft	Sulphide	4212
SOFTSULF	GWKLGSSU	Greywacke	Low grade	Soft	Sulphide	4213
HARDOXID	GWASTHOX	Greywacke	Waste	Hard	Oxide	4300
HARDOXID	GWKHGHOX	Greywacke	High Grade	Hard	Oxide	4312
HARDOXID	GWKLGHOX	Greywacke	Low grade	Hard	Oxide	4313
HARDSULF	GWASTHSU	Greywacke	Waste	Hard	Sulphide	4400
HARDSULF	GWKHGHSU	Greywacke	High Grade	Hard	Sulphide	4412
HARDSULF	GWKLGHSU	Greywacke	Low grade	Hard	Sulphide	4413
	Dumps					8
	Air					500

DENSITY :Density field
NUMSAM:Number of samples used during estimation
ESTVAR:Estimation variance
KVFLAG:Flag that denotes for which blocks (from the waste/hanging wall/ne trend domains), grades were reset to absent because the kriging variance in a particular block was poor (higher than 0.185) & poorly informed by data.

Morila

1	Oxide Ore
2	Transitional Ore
3	Vertical Sulphide
4	LG Horizontal Sulphide
5	HG Horizontal Sulphide
6	Eastern Margin
7	MSZ Extension
9	HWFW Oxide
10	HWFW Transitional
11	HWFW Sulphides
12	Granodiorite
13	Tonalite

KZONE
1 = Oxide ore
2 = Transitional ore
3 = HW

CLASS
1 = Measured
2 = Indicated
3 = Inferred
9 = No Class

ROCKTYPE	Description	Wireframe	Density	KZONE
OXIDE	Above oxide/transitional wireframe	Oxtr07 tr/pt	1.69	1 & 9
TRANSITIONAL	Between oxide/transitional wireframe & transitional/sulphide contact	Trsu07 tr/pt	2.34	2 & 10
SULPHIDE	Below transitional/sulphide contact	Transu tr/pt	2.78	3 to 7
GRANODIORITE	All Material within the Granodiorite wireframe	Granod tr/pt	2.66	12
TONALITE	All Material within the Tonalite wireframe	tn0907 tr/pt	2.66	13

Appendix 2: Model checking and preparation macros

Macros for checking models before simulation

Macro name: model_check.mac

```
!START 1

!COPY     &IN(ny11moda),&OUT(_XX2),AU>0.82<+

!PITMOD   &WIRETR(bp2010_c9tr),&WIREPT(bp2010_c9pt),
&MODELIN(_xx2),&MODELOU(XX2),&RESULTS(XX2),*F1(AU),*F2(KZONE),*F3(DENSITY),*DENSITY(
DENSITY),@DENSITY=1.0,@XSUBCELL=1.0,@YSUBCELL=1.0,@RESOL=0.0

!PITMOD&WIRETR(bp2010_c9tr),&WIREPT(bp2010_c9pt),
&MODELIN(ny0312md),&MODELOU(_XX),&RESULTS(XX),*F1(AU),
*F2(DENSITY),*DENSITY(DENSITY),@DENSITY=1.0,@XSUBCELL=1.0,
@YSUBCELL=1.0,@RESOL=0.0

!END
```

Macro to prepare models after simulation

Macro name: model_prep.mac

```
!START 1

!SELCOP&IN(ge_30),&OUT(ausm1),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),F5(XINC),*F6(YINC),*F7(ZINC),*F
8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),*F14(AU_Simu1),*F15(AU_Simu
2),*F16(AU_Simu3),*F17(AU_Simu4),*F18(AU_Simu5),*F19(AU_Simu6),*F20(AU_Simu7),*F21(AU_Simu
8),*F22(AU_Simu9),*F23(AU_Si10),@KEEPALL=0.0

!EXTRA    &IN(ausm1),&OUT(_ausm1),@APPROX=0.0
'1'=AU_Simu1
'2'=AU_Simu2
'3'=AU_Simu3
'4'=AU_Simu4
'5'=AU_Simu5
'6'=AU_Simu6
'7'=AU_Simu7
'8'=AU_Simu8
'9'=AU_Simu9
'10'=AU_Si10
erase(AU_Simu1)
erase(AU_Simu2)
erase(AU_Simu3)
erase(AU_Simu4)
erase(AU_Simu5)
erase(AU_Simu6)
erase(AU_Simu7)
erase(AU_Simu8)
erase(AU_Simu9)
erase(AU_Si10)
GO
!SELCOP&IN(Ge_30),&OUT(ausm2),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZINC),*
F8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),
*F14(AU_Si11),*F15(AU_Si12),*F16(AU_Si13),*F17(AU_Si14),*F18(AU_Si15),*F19(AU_Si16),*F20(AU_S
i17),*F21(AU_Si18),*F22(AU_Si19),*F23(AU_Si20),@KEEPALL=0.0

!EXTRA    &IN(ausm2),&OUT(_ausm2),@APPROX=0.0
'11'=AU_Si11
'12'=AU_Si12
'13'=AU_Si13
'14'=AU_Si14
'15'=AU_Si15
'16'=AU_Si16
```

```
'17'=AU_Si17
'18'=AU_Si18
'19'=AU_Si19
'20'=AU_Si20
erase(AU_Si11)
erase(AU_Si12)
erase(AU_Si13)
erase(AU_Si14)
erase(AU_Si15)
erase(AU_Si16)
erase(AU_Si17)
erase(AU_Si18)
erase(AU_Si19)
erase(AU_Si20)
GO

!SELCOP&IN(Ge_30),&OUT(ausm3),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZINC),*
F8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),*F14(AU_Si21),*F15(AU_Si22),
*F16(AU_Si23),*F17(AU_Si24),*F18(AU_Si25),*F19(AU_Si26),*F20(AU_Si27),*F21(AU_Si28),
*F22(AU_Si29),*F23(AU_Si30),@KEEPALL=0.0

!EXTRA    &IN(ausm3),&OUT(_ausm3),@APPROX=0.0
'21'=AU_Si21
'22'=AU_Si22
'23'=AU_Si23
'24'=AU_Si24
'25'=AU_Si25
'26'=AU_Si26
'27'=AU_Si27
'28'=AU_Si28
'29'=AU_Si29
'30'=AU_Si30
erase(AU_Si21)
erase(AU_Si22)
erase(AU_Si23)
erase(AU_Si24)
erase(AU_Si25)
erase(AU_Si26)
erase(AU_Si27)
erase(AU_Si28)
erase(AU_Si29)
erase(AU_Si30)
GO

!SELCOP   &IN(Ge_60),&OUT(ausm4),*F1(IJK),*F2(XC),*F3(YC),
*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZINC),*F8(XMORIG),
*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),*F14(AU_Si31),*F15(AU_Si32),*F16(AU_Si3
3),*F17(AU_Si34),*F18(AU_Si35),*F19(AU_Si36),*F20(AU_Si37),*F21(AU_Si38),*F22(AU_Si39),*F23(AU
_Si40),@KEEPALL=0.0

!EXTRA    &IN(ausm4),&OUT(_ausm4),@APPROX=0.0
'31'=AU_Si31
'32'=AU_Si32
'33'=AU_Si33
'34'=AU_Si34
'35'=AU_Si35
'36'=AU_Si36
'37'=AU_Si37
'38'=AU_Si38
'39'=AU_Si39
'40'=AU_Si40
erase(AU_Si31)
erase(AU_Si32)
erase(AU_Si33)
erase(AU_Si34)
erase(AU_Si35)
erase(AU_Si36)
```

```
erase(AU_Si37)
erase(AU_Si38)
erase(AU_Si39)
erase(AU_Si40)
GO

!SELCOP&IN(Ge_60),&OUT(ausm5),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZINC),*
F8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),
*F14(AU_Si41),*F15(AU_Si42),*F16(AU_Si43),*F17(AU_Si44),*F18(AU_Si45),*F19(AU_Si46),*F20(AU_S
i47),*F21(AU_Si48),*F22(AU_Si49),*F23(AU_Si50),@KEEPALL=0.0

!EXTRA    &IN(ausm5),&OUT(_ausm5),@APPROX=0.0
'41'=AU_Si41
'42'=AU_Si42
'43'=AU_Si43
'44'=AU_Si44
'45'=AU_Si45
'46'=AU_Si46
'47'=AU_Si47
'48'=AU_Si48
'49'=AU_Si49
'50'=AU_Si50
erase(AU_Si41)
erase(AU_Si42)
erase(AU_Si43)
erase(AU_Si44)
erase(AU_Si45)
erase(AU_Si46)
erase(AU_Si47)
erase(AU_Si48)
erase(AU_Si49)
erase(AU_Si50)
GO

!SELCOP&IN(Ge_60),&OUT(ausm6),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZINC),*
F8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),*F14(AU_Si51),*F15(AU_Si52),
*F16(AU_Si53),*F17(AU_Si54),*F18(AU_Si55),*F19(AU_Si56),*F20(AU_Si57),*F21(AU_Si58),*F22(AU_S
i59),*F23(AU_Si60),@KEEPALL=0.0
!EXTRA    &IN(ausm6),&OUT(_ausm6),@APPROX=0.0
'51'=AU_Si51
'52'=AU_Si52
'53'=AU_Si53
'54'=AU_Si54
'55'=AU_Si55
'56'=AU_Si56
'57'=AU_Si57
'58'=AU_Si58
'59'=AU_Si59
'60'=AU_Si60
erase(AU_Si51)
erase(AU_Si52)
erase(AU_Si53)
erase(AU_Si54)
erase(AU_Si55)
erase(AU_Si56)
erase(AU_Si57)
erase(AU_Si58)
erase(AU_Si59)
erase(AU_Si60)
GO
!SELCOP&IN(Ge_100),&OUT(ausm7),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZINC)
,*F8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),*F14(AU_Si61),*F15(AU_Si62
),*F16(AU_Si63),*F17(AU_Si64),*F18(AU_Si65),*F19(AU_Si66),*F20(AU_Si67),*F21(AU_Si68),
*F22(AU_Si69),*F23(AU_Si70),@KEEPALL=0.0

!EXTRA    &IN(ausm7),&OUT(_ausm7),@APPROX=0.0
'61'=AU_Si61
```

```
'62'=AU_Si62
'63'=AU_Si63
'64'=AU_Si64
'65'=AU_Si65
'66'=AU_Si66
'67'=AU_Si67
'68'=AU_Si68
'69'=AU_Si69
'70'=AU_Si70
erase(AU_Si61)
erase(AU_Si62)
erase(AU_Si63)
erase(AU_Si64)
erase(AU_Si65)
erase(AU_Si66)
erase(AU_Si67)
erase(AU_Si68)
erase(AU_Si69)
erase(AU_Si70)
GO

!SELCOP&IN(Ge_100),&OUT(ausm8),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZINC)
,*F8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),*F14(AU_Si71),*F15(AU_Si72
),*F16(AU_Si73),*F17(AU_Si74),*F18(AU_Si75),*F19(AU_Si76),*F20(AU_Si77),*F21(AU_Si78),*F22(AU_
Si79),*F23(AU_Si80),@KEEPALL=0.0

!EXTRA    &IN(ausm8),&OUT(_ausm8),@APPROX=0.0
'71'=AU_Si71
'72'=AU_Si72
'73'=AU_Si73
'74'=AU_Si74
'75'=AU_Si75
'76'=AU_Si76
'77'=AU_Si77
'78'=AU_Si78
'79'=AU_Si79
'80'=AU_Si80
erase(AU_Si71)
erase(AU_Si72)
erase(AU_Si73)
erase(AU_Si74)
erase(AU_Si75)
erase(AU_Si76)
erase(AU_Si77)
erase(AU_Si78)
erase(AU_Si79)
erase(AU_Si80)
GO

!SELCOP&IN(Ge_100),&OUT(ausm9),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZINC)
,*F8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),*F14(AU_Si81),*F15(AU_Si82
),*F16(AU_Si83),*F17(AU_Si84),*F18(AU_Si85),*F19(AU_Si86),*F20(AU_Si87),*F21(AU_Si88),*F22(AU_
Si89),*F23(AU_Si90),@KEEPALL=0.0

!EXTRA    &IN(ausm9),&OUT(_ausm9),@APPROX=0.0
'81'=AU_Si81
'82'=AU_Si82
'83'=AU_Si83
'84'=AU_Si84
'85'=AU_Si85
'86'=AU_Si86
'87'=AU_Si87
'88'=AU_Si88
'89'=AU_Si89
'90'=AU_Si90
erase(AU_Si81)
erase(AU_Si82)
```

```
erase(AU_Si83)
erase(AU_Si84)
erase(AU_Si85)
erase(AU_Si86)
erase(AU_Si87)
erase(AU_Si88)
erase(AU_Si89)
erase(AU_Si90)
GO

!SELCOP&IN(Ge_100),&OUT(ausm10),*F1(IJK),*F2(XC),*F3(YC),*F4(ZC),*F5(XINC),*F6(YINC),*F7(ZIN
C),*F8(XMORIG),*F9(YMORIG),*F10(ZMORIG),*F11(NX),*F12(NY),*F13(NZ),*F14(AU_Si91),*F15(AU_Si
92),*F16(AU_Si93),*F17(AU_Si94),*F18(AU_Si95),*F19(AU_Si96),*F20(AU_Si97),*F21(AU_Si98),*F22(A
U_Si99),*F23(AU_S100),@KEEPALL=0.0

!EXTRA    &IN(ausm10),&OUT(_ausm10),@APPROX=0.0
'91'=AU_Si91
'92'=AU_Si92
'93'=AU_Si93
'94'=AU_Si94
'95'=AU_Si95
'96'=AU_Si96
'97'=AU_Si97
'98'=AU_Si98
'99'=AU_Si99
'100'=AU_S100
erase(AU_Si91)
erase(AU_Si92)
erase(AU_Si93)
erase(AU_Si94)
erase(AU_Si95)
erase(AU_Si96)
erase(AU_Si97)
erase(AU_Si98)
erase(AU_Si99)
erase(AU_S100)
GO

!END
```

Macros for Whittle inputs preparation

Macro Name:Model_prep_wht.mac

```
!START 1
!echo **Note the Simulated model has no subcell ( XINC,YINC & ZINC are implicit)**
!echo **The simulated model block Size is 10X10x3  not best of optimisation******
!echo **A prot is created every thing the same except the ZINC & NZ ******
!echo **The new proto is 10X10x3   best of optimisation******

!PROTOM   &OUT(prot),@ROTMOD=0.0
n
n
49060
9300
650
10
10
10
420
272
70

!echo ** Each Simulated Model is regulirized  to the new proto******

!REGMOD   &IN1(prot),&IN2(_ausm1),&OUT(ausim1a),*F1(1),*F2(2),
```

```
        *F3(3),*F4(4),*F5(5),*F6(6),*F7(7),*F8(8),*F9(9),*F10(10)

!REGMOD   &IN1(prot),&IN2(_ausm2),&OUT(ausim2a),*F1(11),*F2(12),
        *F3(13),*F4(14),*F5(15),*F6(16),*F7(17),*F8(18),*F9(19),
        *F10(20)

!REGMOD   &IN1(prot),&IN2(_ausm3),&OUT(ausim3a),*F1(21),*F2(22),
        *F3(23),*F4(24),*F5(25),*F6(26),*F7(27),*F8(28),*F7(29),*F8(30)


!REGMOD   &IN1(prot),&IN2(_ausm4),&OUT(ausim4a),*F1(31),*F2(32),
             *F3(33),*F4(34),*F5(35),*F6(36),*F7(37),*F8(38),*F9(39),
        *F10(40)

!REGMOD   &IN1(prot),&IN2(_ausm5),&OUT(ausim5a),*F1(41),*F2(42),
        *F3(43),*F4(44),*F5(45),*F6(46),*F7(47),*F8(48),*F9(49),
        *F10(50)

!REGMOD   &IN1(prot),&IN2(_ausm6),&OUT(ausim6a),*F1(51),*F2(52),
        *F3(53),*F4(54),*F5(55),*F6(56),*F7(57),*F8(58),*F9(59),
        *F10(60)

!REGMOD   &IN1(prot),&IN2(_ausm7),&OUT(ausim7a),*F1(61),*F2(62),
        *F3(63),*F4(64),*F5(65),*F6(66),*F7(67),*F8(68),*F9(69),
        *F10(70)

!REGMOD   &IN1(prot),&IN2(_ausm8),&OUT(ausim8a),*F1(71),*F2(72),
        *F3(73),*F4(74),*F5(75),*F6(76),*F7(77),*F8(78),*F9(79),
        *F10(80)

!REGMOD   &IN1(prot),&IN2(_ausm9),&OUT(ausim9a),*F1(81),*F2(82),
        *F3(83),*F4(84),*F5(85),*F6(86),*F7(87),*F8(88),*F9(89),
        *F10(90)

!REGMOD   &IN1(prot),&IN2(_ausm10),&OUT(ausim10a),*F1(91),*F2(92),
        *F3(93),*F4(94),*F5(95),*F6(96),*F7(97),*F8(98),*F9(99),
        *F10(100)

!echo ** all 10x10x10 Simulated Models are combine to on efile Ausim******

!ADDMOD   &IN1(ausim1a),&IN2(ausim2a),&OUT(tmp1),@TOLERNCE=0.001

!DELETE   &IN(ausim1a),@CONFIRM=0.0
!DELETE   &IN(ausim2a),@CONFIRM=0.0

!ADDMOD   &IN1(tmp1),&IN2(ausim3a),&OUT(tmp2),@TOLERNCE=0.001
!ADDMOD   &IN1(tmp2),&IN2(ausim4a),&OUT(tmp1),@TOLERNCE=0.001

!DELETE   &IN(ausim3a),@CONFIRM=0.0
!DELETE   &IN(ausim4a),@CONFIRM=0.0

!ADDMOD   &IN1(tmp1),&IN2(ausim5a),&OUT(tmp2),@TOLERNCE=0.001
!ADDMOD   &IN1(tmp2),&IN2(ausim6a),&OUT(tmp1),@TOLERNCE=0.001

!DELETE   &IN(ausim5a),@CONFIRM=0.0
!DELETE   &IN(ausim6a),@CONFIRM=0.0


!ADDMOD   &IN1(tmp1),&IN2(ausim7a),&OUT(tmp2),@TOLERNCE=0.001
!ADDMOD   &IN1(tmp2),&IN2(ausim8a),&OUT(tmp1),@TOLERNCE=0.001
!DELETE   &IN(ausim7a),@CONFIRM=0.0
!DELETE   &IN(ausim8a),@CONFIRM=0.0
!ADDMOD   &IN1(tmp1),&IN2(ausim9a),&OUT(tmp2),@TOLERNCE=0.001
!ADDMOD   &IN1(tmp2),&IN2(ausim10a),&OUT(ausim),@TOLERNCE=0.001

!DELETE   &IN(ausim9a),@CONFIRM=0.0
!DELETE   &IN(ausim10a),@CONFIRM=0.0
```

```
!DELETE   &IN(tmp1),@CONFIRM=0.0
!DELETE   &IN(tmp2),@CONFIRM=0.0

!echo ** Important files from the ny0312md are selected ******

!INPFIL   &OUT(Field)

FIELDNAM
A
8
Y

$
Y

IJK
XC
YC
ZC
XINC
YINC
ZINC
DENSITY
AU
OXIDE
MZONE
KZONE
XMORIG
YMORIG
ZMORIG
NX
NY
NZ
!

!SELCOPY   &IN(ny0312md),&OUT(tem1),&FIELDLST(Field),*F1(IJK),
           @KEEPALL=1.0

!echo **Assigning  MCAF **
!echo ** Note: for this study we assume the model is not mined such as no depletion & waste dump
addition **
!echo **Note MCAF are calculated based on Cutback field for other blocks  from Geita Script such as
WST=1 **

!EXTRA    &IN(tem1),&OUT(_1)

 if (ZC>=1310)
  O_MINS = -0.0575*ZC + 86.199
  W_MINS = -0.000320*RAIS(ZC,2) + 0.893161*ZC - 604.0337
 elseif (ZC<1310 & ZC>=1270)
  O_MINS = -0.0575*ZC + 86.199
  W_MINS = 0.0095*ZC + 0.6765
 else
  O_MINS = -0.0575*ZC + 86.199
  W_MINS = -0.0824*ZC + 115.17
 end

if(W_MINS<2) W_MINS=2 END
if(O_MINS<2) O_MINS=2 END
GO
!EXTRA    &IN(_1),&OUT(_2)
BCM=XINC*YINC*ZINC
if(OXIDE==2)
 TOTO_MINS=(4.94+O_MINS+1.5)*(1+5/100)
 TOTW_MINS=(4.48+W_MINS+1.5)*(1+5/100)
 TOTB_MINS=(3.5+W_MINS+1.8)*(1+5/100)
 TRO_REQD= (4.94+O_MINS+1.5)/4.94
```

```
 TRW_REQD= (4.48+W_MINS+1.5)/4.48
 TRB_REQD= (3.5+W_MINS+1.8)/3.5
 TRO_BCMH= (54/1/TOTO_MINS)*130/DENSITY
 TRW_BCMH= (54/1/TOTW_MINS)*130/DENSITY
 TRB_BCMH= (54/1/TOTB_MINS)*190/DENSITY
elseif(OXIDE==1)
 TOTO_MINS=(5.16+O_MINS+1.5)*(1+5/100)
 TOTW_MINS=(4.73+W_MINS+1.5)*(1+5/100)
 TOTB_MINS=(3.83+W_MINS+1.8)*(1+5/100)
 TRO_REQD= (5.16+O_MINS+1.5)/5.16
 TRW_REQD= (4.73+W_MINS+1.5)/4.73
 TRB_REQD= (3.83+W_MINS+1.8)/3.83
 TRO_BCMH= (54/1/TOTO_MINS)*130/DENSITY
 TRW_BCMH= (54/1/TOTW_MINS)*130/DENSITY
 TRB_BCMH= (54/1/TOTB_MINS)*190/DENSITY
else
 TOTO_MINS=(5.9+O_MINS+1.5)*(1+5/100)
 TOTW_MINS=(5.26+W_MINS+1.5)*(1+5/100)
 TOTB_MINS=(3.95+W_MINS+1.8)*(1+5/100)
 TRO_REQD= (5.9+O_MINS+1.5)/5.9
 TRW_REQD= (5.26+W_MINS+1.5)/5.26
 TRB_REQD= (3.95+W_MINS+1.8)/3.95
 TRO_BCMH= (54/1/TOTO_MINS)*130/DENSITY
 TRW_BCMH= (54/1/TOTW_MINS)*130/DENSITY
 TRB_BCMH= (54/1/TOTB_MINS)*190/DENSITY
end
TRKO_BCM=176.47/TRO_BCMH
TRKW_BCM=176.47/TRW_BCMH
TRKB_BCM=239.56/TRB_BCMH
GO

!EXTRA    &IN(_2),&OUT(tem2)
TEMP=0.6+0.17+0.76+0.08+0.32

if(OXIDE==2)
  COSTO_BCM=TRKO_BCM+0.62+TEMP+0.61
  COSTW_BCM=TRKW_BCM+0.65+TEMP+0.61
  T_LOAD=0.65
 end

if(OXIDE==1)

  COSTO_BCM=TRKO_BCM+0.69+TEMP+0.8
  COSTW_BCM=TRKW_BCM+0.71+TEMP+0.8
  T_LOAD=0.71
  end

if(OXIDE==0)
  COSTO_BCM=TRKO_BCM+0.83+TEMP+1.78+0
  COSTW_BCM=TRKW_BCM+0.82+TEMP+1.12+0
  end

MCAF=COSTW_BCM/DENSITY
ORE_INC=(COSTO_BCM/DENSITY)-MCAF

GO
!DELETE   &IN(_1),@CONFIRM=0.0
!DELETE   &IN(_2),@CONFIRM=0.0

!echo **PCAF **
!EXTRA    &IN(tem2),&OUT(tem3)
PCAF=1
TEMP=9.54+0+2.33+0.76
if(OXIDE==2)
 PCAF=(15.57+TEMP+ORE_INC)/(15.57+TEMP)
elseif(OXIDE==1)
 PCAF=(14.91+TEMP+ORE_INC)/(14.91+TEMP)
```

```
else
 PCAF=(16.15+TEMP+ORE_INC)/(16.15+TEMP)
end

GO
!echo **The Resource model is regulirized  to the new proto with AU ,OXIDE , DENSITY, MCAF &
PCAF******

!REGMOD   &IN1(prot),&IN2(tem3),&OUT(t2),*F1(AU),*F2(OXIDE),
                    *F3(DENSITY),*F4(PCAF),*F5(MCAF),*F6(MZONE)

!DELETE   &IN(tem1),@CONFIRM=0.0
!DELETE   &IN(tem2),@CONFIRM=0.0
!DELETE   &IN(tem3),@CONFIRM=0.0

!echo **Assigning Whittle Rock NAMES, GTZONE **
!echo **Calculating Whittle Block tonnage (TON), & Pacel tonnage (RTON) **

!SELEXY   &IN(t2),&PERIM(gzonest),&OUT(t3),*X(XC),*Y(YC),
       *ATTRIB1(GZONE),@OUTSIDE=0.0

!EXTRA    &IN(t3),&OUT(t4)

IF(OXIDE <0.5) OXIDE=0 END
IF(OXIDE >=0.5 & OXIDE <1.5) OXIDE=1 END
IF(OXIDE>=1.5 ) OXIDE=2 END

TYPE;a4 = "SUL"

IF(OXIDE==2) TYPE="OXI" END
IF(OXIDE==1) TYPE="TRA" END
IF(OXIDE==0) TYPE="SUL" END
IF(MZONE==0) TYPE="WST" END

TON=XINC*YINC*ZINC*DENSITY
RTON=TON
IF(OXIDE==2 & GZONE==1) GTZONE=1 END
IF(OXIDE==1 & GZONE==1) GTZONE=2 END
IF(OXIDE==0 & GZONE==1) GTZONE=3 END
IF(OXIDE==2 & GZONE==2) GTZONE=4 END
IF(OXIDE==1 & GZONE==2) GTZONE=5 END
IF(OXIDE==0 & GZONE==2) GTZONE=6 END
IF(OXIDE<=1140 & GZONE==3) GTZONE=7 END
IF(OXIDE<=1150 & GZONE==3) GTZONE=8 END
IF(OXIDE<=950 & GZONE==3) GTZONE=9 END

GO

echo **The ausim Simulated model & the regulirized Resource model are combined***

!ADDMOD   &IN1(ausim),&IN2(t4),&OUT(t5),@TOLERNCE=0.001

echo **copy blocks within the Resource Model***

!COPY     &IN(t5),&OUT(_mod1),DENSITY>0.1

!DELETE   &IN(t2),@CONFIRM=0.0
!DELETE   &IN(t3),@CONFIRM=0.0
!DELETE   &IN(t4),@CONFIRM=0.0
!DELETE   &IN(t5),@CONFIRM=0.0
!echo **Generating Wittle Model & parameter files for all Simulations **

!field $EXIST#=_mod1,$recs#=0,$AU1#=1
!LET $AU1#=17
!LET $AUMAX#=100

!LOOP1:REM
```

```
!LET $AU1#=$AU1#+1
!IF $AU1# > $AUMAX#, GOTO LOOPEND
!LET $AU#={int($AU1#)}

!INPFIL   &OUT(Field)

FIELDNAM
A
8
Y

$
Y

IJK
XC
YC
ZC
XINC
YINC
ZINC
RTON
MCAF
PCAF
TON
$AU#
TYPE
GTZONE
XMORIG
YMORIG
ZMORIG
NX
NY
NZ
!

!SELCOPY   &IN(_mod1),&OUT(_mod2),&FIELDLST(Field),*F1(IJK),
           @KEEPALL=1.0


!EXTRA    &IN(_mod2),&OUT(_mod3),@APPROX=0.0
AU=$AU#
AUMET=TON*AU
IF (AUMET==absent()) AUMET=0 END

GO
!FXOUT    &IN(_mod3),@TOLTON=0.5,@FORMAT=0.0,@ELEMENT=1.0,
         @ZONEFLD=1.0
$AU#.par
$AU#.mod
B
999
0.0
n
RTON
MCAF
PCAF
GTZONE
TON
AUMET
TYPE
Y

!GOTO LOOP1
!LOOPEND:REM
!RETURN
!END
```

Appendix 3: Grade- tonnage curve data for Morila and CVSA Mines

Morila Mine

All Rock types / All CLASS			
Cutoff g/t	Ton SMU (kton)	Metal (kg)	Grade SMU (g/t)
≥ 0.0	1,734,545	323,100	0.19
≥ 0.4	141,533	323,100	2.28
≥ 0.5	118,533	312,813	2.64
≥ 0.7	93,188	297,884	3.20
≥ 1.0	73,855	281,674	3.81
≥ 1.1	69,075	276,656	4.01
≥ 1.2	64,765	271,685	4.19
≥ 1.3	60,881	266,815	4.38
≥ 1.4	57,423	262,150	4.57
≥ 2.0	41,344	235,110	5.69
≥ 2.5	32,413	215,105	6.64
≥ 3.0	25,963	197,372	7.60
≥ 3.5	20,999	181,254	8.63
≥ 4.0	17,438	167,898	9.63
≥ 4.5	14,749	156,465	10.61
≥ 5.0	12,671	146,573	11.57
≥ 7.0	8,068	119,414	14.80
≥ 10.0	4,660	90,948	19.52

All Rock types / All CLASS			
Cutoff g/t	Ton1 (kton)	Metal (kg)	Grade1 (g/t)
≥ 0.0			
≥ 0.4	55,773	307,869	5.52
≥ 0.5	54,482	307,277	5.64
≥ 0.7	51,506	305,945	5.94
≥ 1.0	46,572	301,319	6.47
≥ 1.1	45,000	299,702	6.66
≥ 1.2	43,487	298,319	6.86
≥ 1.3	41,920	296,374	7.07
≥ 1.4	40,386	294,011	7.28
≥ 2.0	32,615	281,141	8.62
≥ 2.5	27,571	269,647	9.78
≥ 3.0	23,764	259,261	10.91
≥ 3.5	20,725	249,324	12.03
≥ 4.0	18,196	240,011	13.19
≥ 4.5	16,128	231,113	14.33
≥ 5.0	14,469	223,253	15.43
≥ 7.0	9,914	196,487	19.82
≥ 10.0	6,485	167,969	25.90

All Rock types / All CLASS			
Cutoff g/t	Ton25 (kton)	Metal (kg)	Grade25 (g/t)
≥ 0.0			
≥ 0.4	55,731	293,702	5.27
≥ 0.5	54,429	292,830	5.38
≥ 0.7	51,513	291,047	5.65
≥ 1.0	46,632	287,251	6.16
≥ 1.1	44,984	285,200	6.34
≥ 1.2	43,432	283,608	6.53
≥ 1.3	41,914	281,659	6.72
≥ 1.4	40,452	279,526	6.91
≥ 2.0	32,762	266,680	8.14
≥ 2.5	27,851	255,669	9.18
≥ 3.0	23,875	244,723	10.25
≥ 3.5	20,650	234,377	11.35
≥ 4.0	18,161	225,011	12.39
≥ 4.5	16,066	216,248	13.46
≥ 5.0	14,424	208,424	14.45
≥ 7.0	9,818	181,245	18.46
≥ 10.0	6,371	152,532	23.94

All Rock types / All CLASS			
Cutoff g/t	Ton50 (kton)	Metal (kg)	Grade50 (g/t)
≥ 0.0			
≥ 0.4	55,633	310,988	5.59
≥ 0.5	54,347	310,321	5.71
≥ 0.7	51,370	308,222	6.00
≥ 1.0	46,456	304,288	6.55
≥ 1.1	44,845	302,702	6.75
≥ 1.2	43,348	300,838	6.94
≥ 1.3	41,864	298,907	7.14
≥ 1.4	40,323	296,777	7.36
≥ 2.0	32,530	283,663	8.72
≥ 2.5	27,637	272,773	9.87
≥ 3.0	23,637	261,661	11.07
≥ 3.5	20,552	251,761	12.25
≥ 4.0	18,067	242,461	13.42
≥ 4.5	15,976	233,566	14.62
≥ 5.0	14,286	225,580	15.79
≥ 7.0	9,935	199,887	20.12
≥ 10.0	6,567	171,912	26.18

All Rock types / All CLASS			
Cutoff g/t	Ton75 (kton)	Metal (kg)	Grade75 (g/t)
≥ 0.0			
≥ 0.4	56,090	320,273	5.71
≥ 0.5	54,903	320,085	5.83
≥ 0.7	51,913	318,227	6.13
≥ 1.0	47,183	314,240	6.66
≥ 1.1	45,693	312,540	6.84
≥ 1.2	44,148	310,802	7.04
≥ 1.3	42,601	308,855	7.25
≥ 1.4	41,189	306,857	7.45
≥ 2.0	33,569	294,064	8.76
≥ 2.5	28,551	282,939	9.91
≥ 3.0	24,694	272,377	11.03
≥ 3.5	21,545	261,991	12.16
≥ 4.0	18,931	252,351	13.33
≥ 4.5	16,765	243,097	14.50
≥ 5.0	14,953	234,459	15.68
≥ 7.0	10,274	206,924	20.14
≥ 10.0	6,745	177,673	26.34

All Rock types / All CLASS			
Cutoff g/t	Ton100 (kton)	Metal (kg)	Grade100 (g/t)
≥ 0.0			
≥ 0.4	55,818	312,025	5.59
≥ 0.5	54,506	311,231	5.71
≥ 0.7	51,500	309,517	6.01
≥ 1.0	46,774	305,436	6.53
≥ 1.1	45,129	303,716	6.73
≥ 1.2	43,556	302,276	6.94
≥ 1.3	41,932	300,236	7.16
≥ 1.4	40,383	298,025	7.38
≥ 2.0	32,599	284,920	8.74
≥ 2.5	27,491	273,260	9.94
≥ 3.0	23,537	262,441	11.15
≥ 3.5	20,436	252,387	12.35
≥ 4.0	18,012	243,337	13.51
≥ 4.5	16,074	235,155	14.63
≥ 5.0	14,413	227,299	15.77
≥ 7.0	9,994	201,274	20.14
≥ 10.0	6,565	172,793	26.32

All Rock types / All CLASS			
Cutoff g/t	TonAU (kton)	Metal (kg)	GradeAU (g/t)
≥ 0.0			
≥ 0.4	156,740	316,615	2.02
≥ 0.5	127,061	304,947	2.40
≥ 0.7	94,113	284,220	3.02
≥ 1.0	71,402	264,903	3.71
≥ 1.1	66,728	260,240	3.90
≥ 1.2	62,470	254,879	4.08
≥ 1.3	58,746	250,258	4.26
≥ 1.4	55,293	246,053	4.45
≥ 2.0	39,602	219,393	5.54
≥ 2.5	31,240	200,870	6.43
≥ 3.0	24,893	183,459	7.37
≥ 3.5	20,178	168,082	8.33
≥ 4.0	16,708	155,052	9.28
≥ 4.5	13,978	143,553	10.27
≥ 5.0	11,937	133,933	11.22
≥ 7.0	7,712	109,051	14.14
≥ 10.0	4,334	81,008	18.69

CVSA Mine

All Rock types / All CLASS			
Cutoff g/t	Ton1 (kton)	Metal (kg)	Grade1 (g/t)
≥ 0.0			
≥ 0.4	4,923	38,250	7.77
≥ 0.5	4,867	38,207	7.85
≥ 0.7	4,760	38,125	8.01
≥ 1.0	4,603	38,017	8.26
≥ 1.1	4,547	37,966	8.35
≥ 1.2	4,481	37,866	8.45
≥ 1.3	4,402	37,765	8.58
≥ 1.4	4,348	37,699	8.67
≥ 2.0	4,054	37,219	9.18
≥ 2.5	3,555	36,087	10.15
≥ 3.0	2,962	34,450	11.63
≥ 3.5	2,529	33,054	13.07
≥ 4.0	2,175	31,733	14.59
≥ 4.5	2,000	31,003	15.50
≥ 5.0	1,825	30,164	16.53
≥ 7.0	1,310	27,111	20.69
≥ 10.0	885	23,576	26.65

All Rock types / All CLASS			
Cutoff g/t	Ton25 (kton)	Metal (kg)	Grade25 (g/t)
≥ 0.0			
≥ 0.4	4,912	37,626	7.66
≥ 0.5	4,866	37,613	7.73
≥ 0.7	4,750	37,526	7.90
≥ 1.0	4,587	37,387	8.15
≥ 1.1	4,535	37,368	8.24
≥ 1.2	4,470	37,276	8.34
≥ 1.3	4,399	37,173	8.45
≥ 1.4	4,348	37,131	8.54
≥ 2.0	4,063	36,651	9.02
≥ 2.5	3,554	35,474	9.98
≥ 3.0	2,971	33,900	11.41
≥ 3.5	2,514	32,404	12.89
≥ 4.0	2,129	30,984	14.55
≥ 4.5	1,962	30,273	15.43
≥ 5.0	1,796	29,487	16.42
≥ 7.0	1,293	26,507	20.50
≥ 10.0	867	22,981	26.50

All Rock types / All CLASS			
Cutoff g/t	Ton50 (kton)	Metal (kg)	Grade50 (g/t)
≥ 0.0			
≥ 0.4	4,929	38,942	7.90
≥ 0.5	4,884	38,925	7.97
≥ 0.7	4,774	38,861	8.14
≥ 1.0	4,618	38,741	8.39
≥ 1.1	4,563	38,696	8.48
≥ 1.2	4,491	38,624	8.60
≥ 1.3	4,421	38,505	8.71
≥ 1.4	4,371	38,463	8.80
≥ 2.0	4,103	37,992	9.26
≥ 2.5	3,583	36,831	10.28
≥ 3.0	3,002	35,246	11.74
≥ 3.5	2,551	33,770	13.24
≥ 4.0	2,174	32,372	14.89
≥ 4.5	2,003	31,632	15.79
≥ 5.0	1,836	30,852	16.80
≥ 7.0	1,327	27,830	20.98
≥ 10.0	908	24,347	26.81

All Rock types / All CLASS			
Cutoff g/t	Ton75 (kton)	Metal (kg)	Grade75 (g/t)
≥ 0.0			
≥ 0.4	4,913	39,749	8.09
≥ 0.5	4,857	39,729	8.18
≥ 0.7	4,750	39,619	8.34
≥ 1.0	4,588	39,507	8.61
≥ 1.1	4,531	39,422	8.70
≥ 1.2	4,459	39,375	8.83
≥ 1.3	4,378	39,268	8.97
≥ 1.4	4,326	39,190	9.06
≥ 2.0	4,038	38,686	9.58
≥ 2.5	3,539	37,585	10.62
≥ 3.0	2,990	36,084	12.07
≥ 3.5	2,584	34,755	13.45
≥ 4.0	2,285	33,652	14.73
≥ 4.5	2,114	32,919	15.57
≥ 5.0	1,937	32,095	16.57
≥ 7.0	1,402	28,921	20.63
≥ 10.0	951	25,182	26.48

All Rock types / All CLASS			
Cutoff g/t	Ton100 (kton)	Metal (kg)	Grade100 (g/t)
≥ 0.0			
≥ 0.4	4,870	36,528	7.50
≥ 0.5	4,808	36,492	7.59
≥ 0.7	4,691	36,447	7.77
≥ 1.0	4,521	36,305	8.03
≥ 1.1	4,462	36,230	8.12
≥ 1.2	4,397	36,144	8.22
≥ 1.3	4,315	36,074	8.36
≥ 1.4	4,261	36,008	8.45
≥ 2.0	3,976	35,502	8.93
≥ 2.5	3,452	34,312	9.94
≥ 3.0	2,863	32,726	11.43
≥ 3.5	2,428	31,295	12.89
≥ 4.0	2,093	30,060	14.36
≥ 4.5	1,912	29,296	15.32
≥ 5.0	1,750	28,530	16.30
≥ 7.0	1,261	25,633	20.32
≥ 10.0	845	22,162	26.22

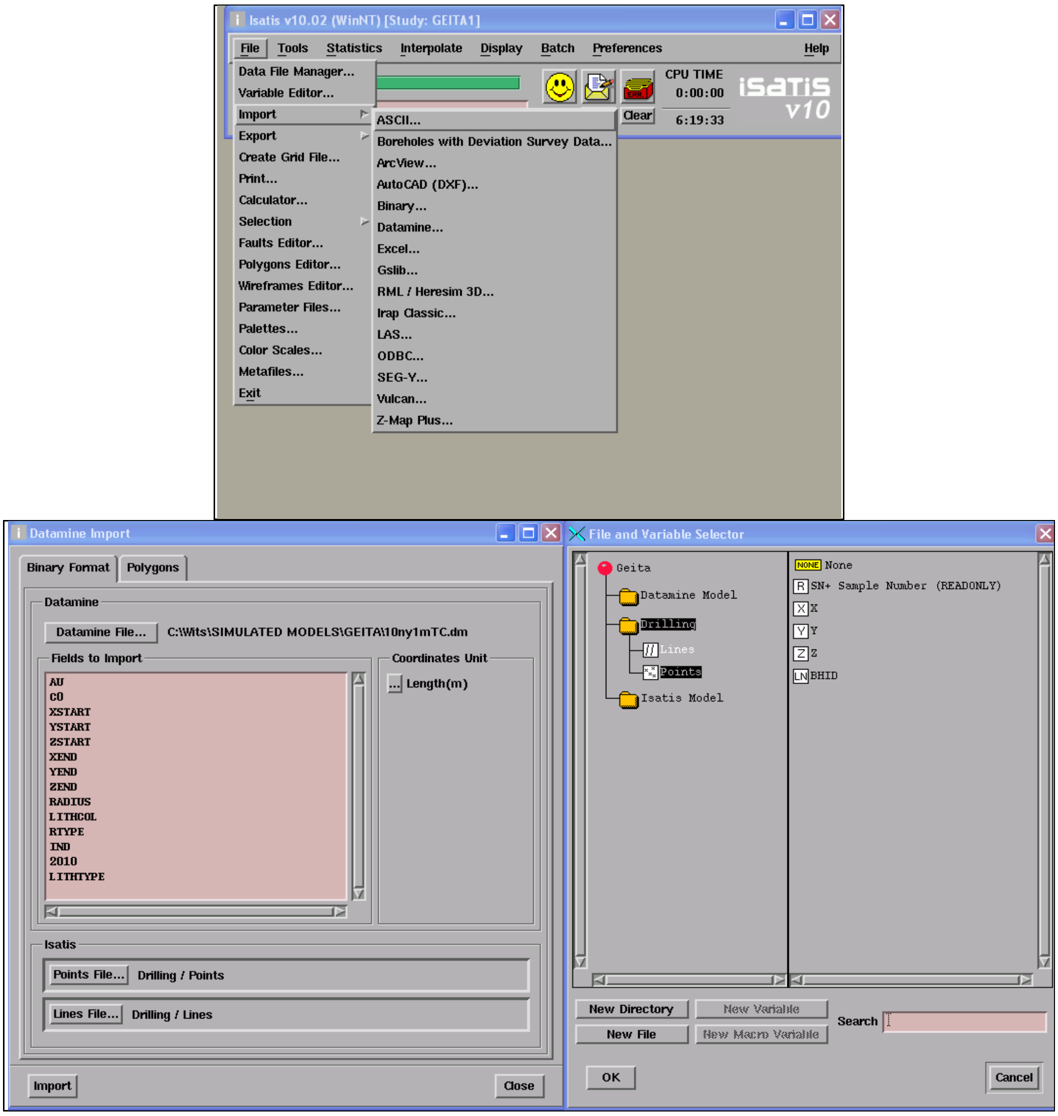
All Rock types / All CLASS			
Cutoff g/t	TonAU (kton)	Metal (kg)	GradeAU (g/t)
≥ 0.0			
≥ 0.4	5,252	23,530	4.48
≥ 0.5	4,528	23,185	5.12
≥ 0.7	3,797	22,742	5.99
≥ 1.0	2,981	22,062	7.40
≥ 1.1	2,776	21,848	7.87
≥ 1.2	2,640	21,704	8.22
≥ 1.3	2,550	21,600	8.47
≥ 1.4	2,481	21,487	8.66
≥ 2.0	2,329	21,237	9.12
≥ 2.5	2,233	21,036	9.42
≥ 3.0	2,094	20,646	9.86
≥ 3.5	1,950	20,181	10.35
≥ 4.0	1,787	19,566	10.95
≥ 4.5	1,658	19,017	11.47
≥ 5.0	1,511	18,312	12.12
≥ 7.0	1,093	15,853	14.51
≥ 10.0	692	12,534	18.11

Appendix 4: Steps in generating simulated models

Importing Datamine drill hole in Isatis

The data needs to be imported into Isatis; main data to be imported are the drill holes and the geological block models. The path to launch is (*File->import->Datamine-->drill hole*).

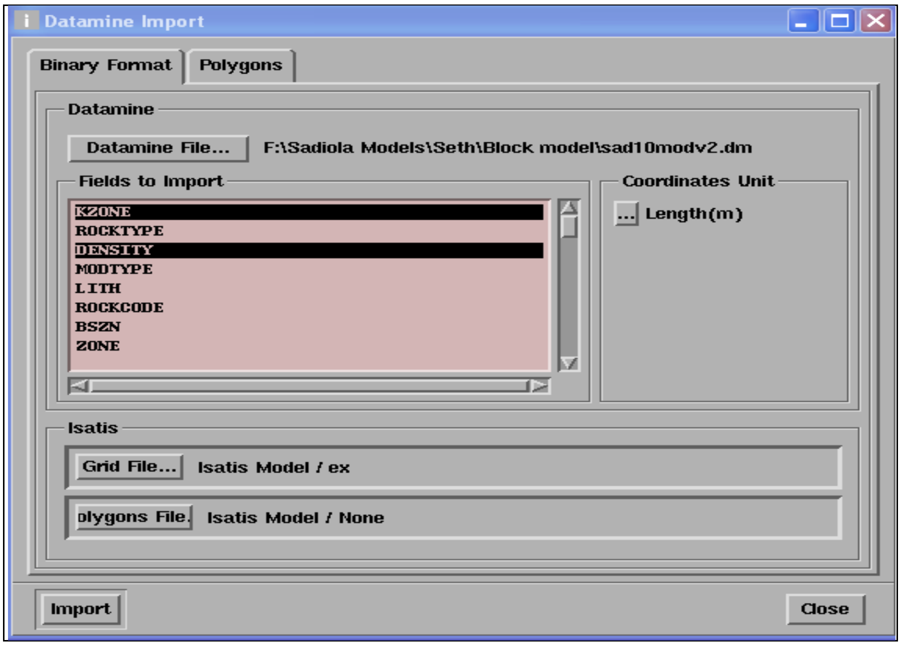
Importing drill hole data window



Importing Datamine block model

Importing a block model in Isatis will have to be imported. The fields to be selected are (KZONE, DENSITY and AU). To import launch (*File->import->Datamine->block model*).

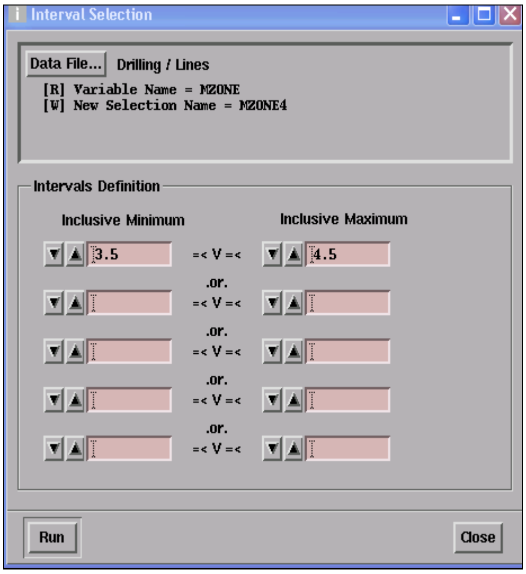
Importing block models window



Creating intervals for selection

After importing the drill holes and the block model, one needs to create a selection of the blocks and drill hole samples in each Kriged zone (KZONE), which will be used for the data analysis and the simulation. This is achieved by launching (*File-> selection-> intervals*).

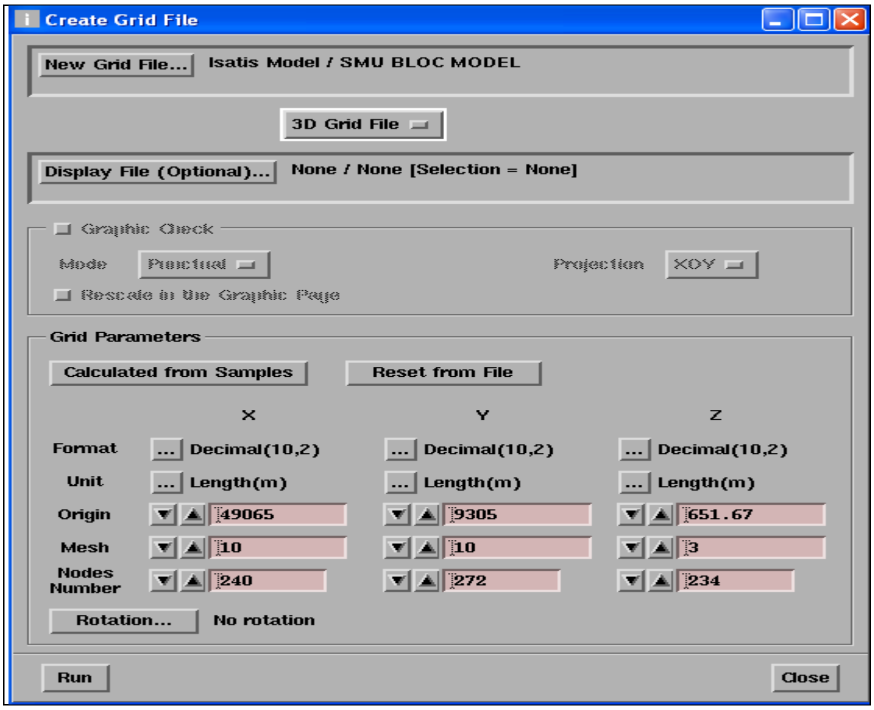
Intervals selection window



Creating Isatis grid

If the Datamine block model contains sub cells, there is a need to create a Isatis 3-D grid but if there are no sub cells in the model, the Datamine block model will be used for the simulation. The difference between the Datamine to Isatis origins is half of the parent cell, which offset to account for the different origin setups. The path to launch is (*File-> create grid file*).

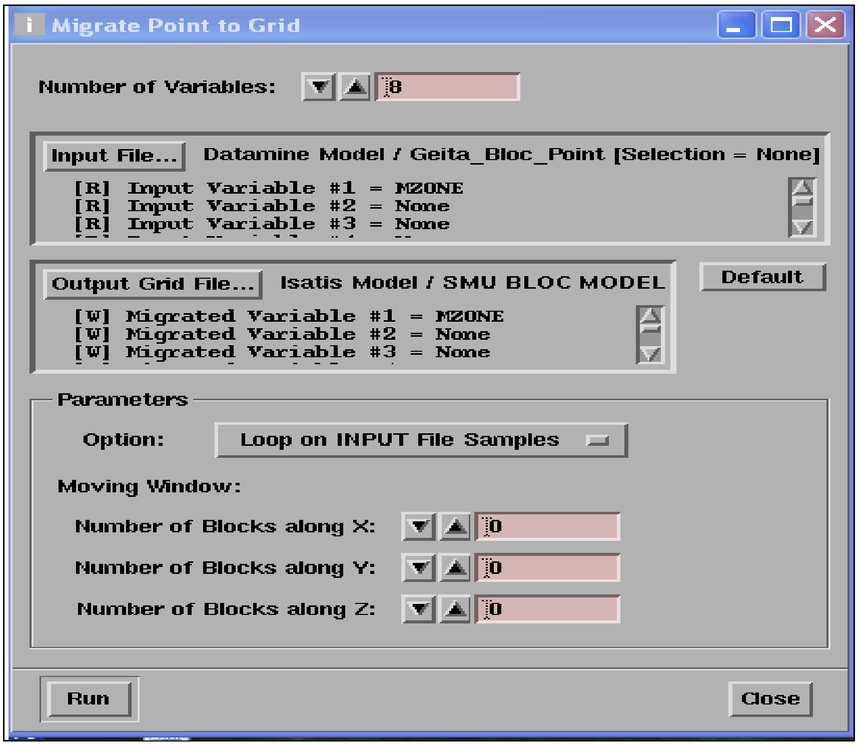
Creating a new grid in Isatis window



Migration of Datamine parameters to Isatis grid

The way to copy the Density and the KZONE parameters or fields from the drill hole data file to Isatis grid file is done with the migration of Datamine parameters to Isatis grid file. There is a need to have the Density and KZONE fields in the Isatis grid in order to do the simulations and to calculate the tonnages. The path is (*Tools->migrate-point-grid*).

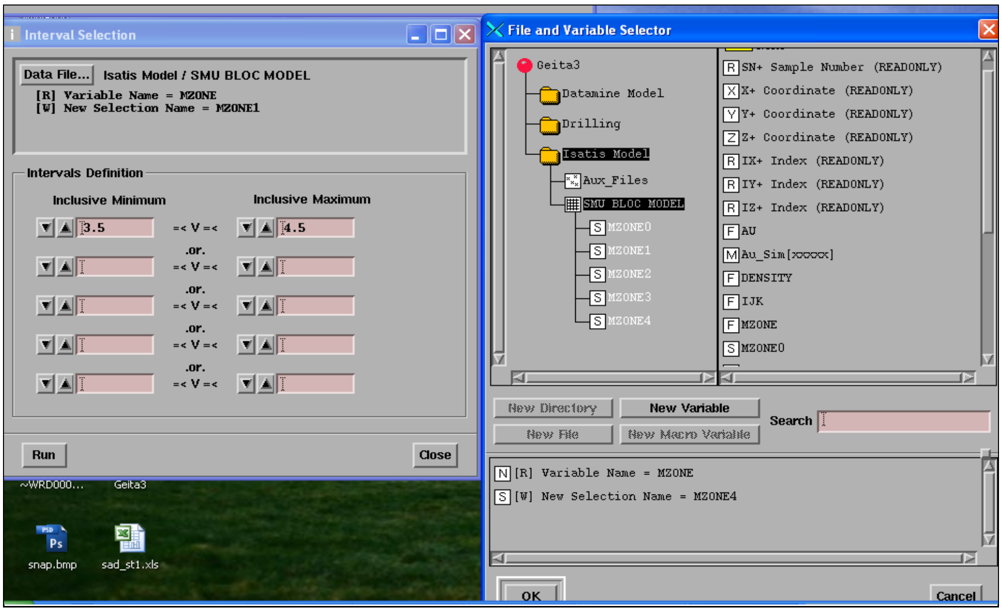
Migrating Datamine parameters to Isatis grid window



Creating selection with Isatis grid

There is the need to make a selection in the Isatis grid, for each KZONE, which will be used for the simulation and the block analysis. To create the selection, the path to launch is (*File-> selection-> intervals*).

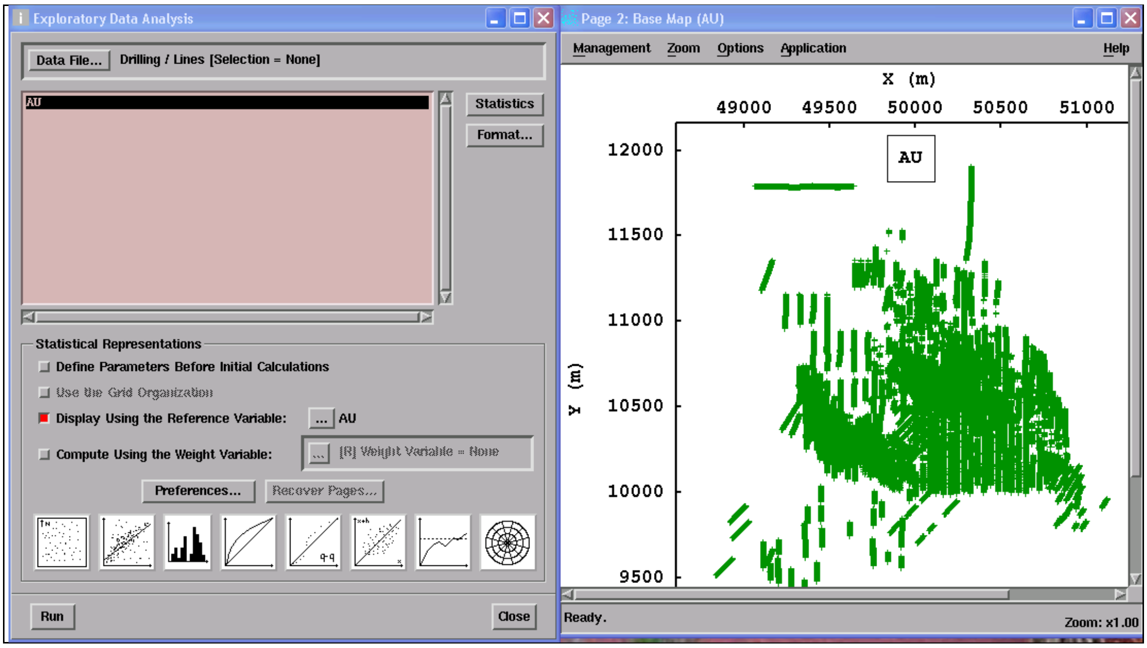
Creation selection with Isatis grid window



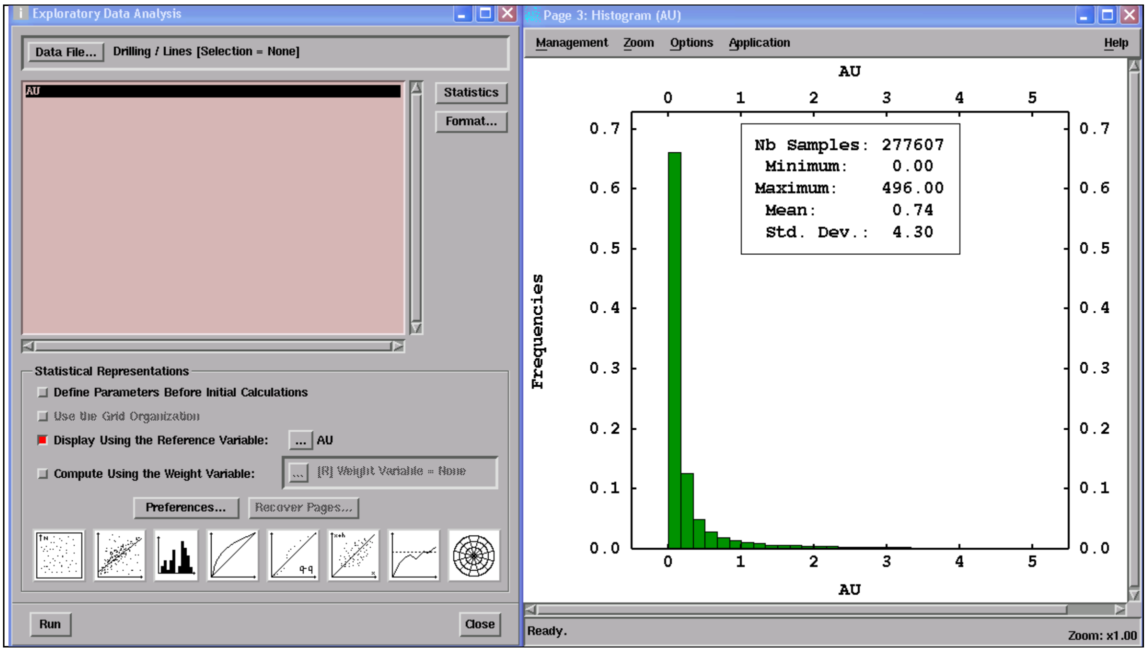
Data analysis (map, histogram)

Data analysis is used to display the maps and histograms to help understand and visualise the data. This is achieved by running (*Statistics->exploratory data analysis*).

Data analysis for maps window



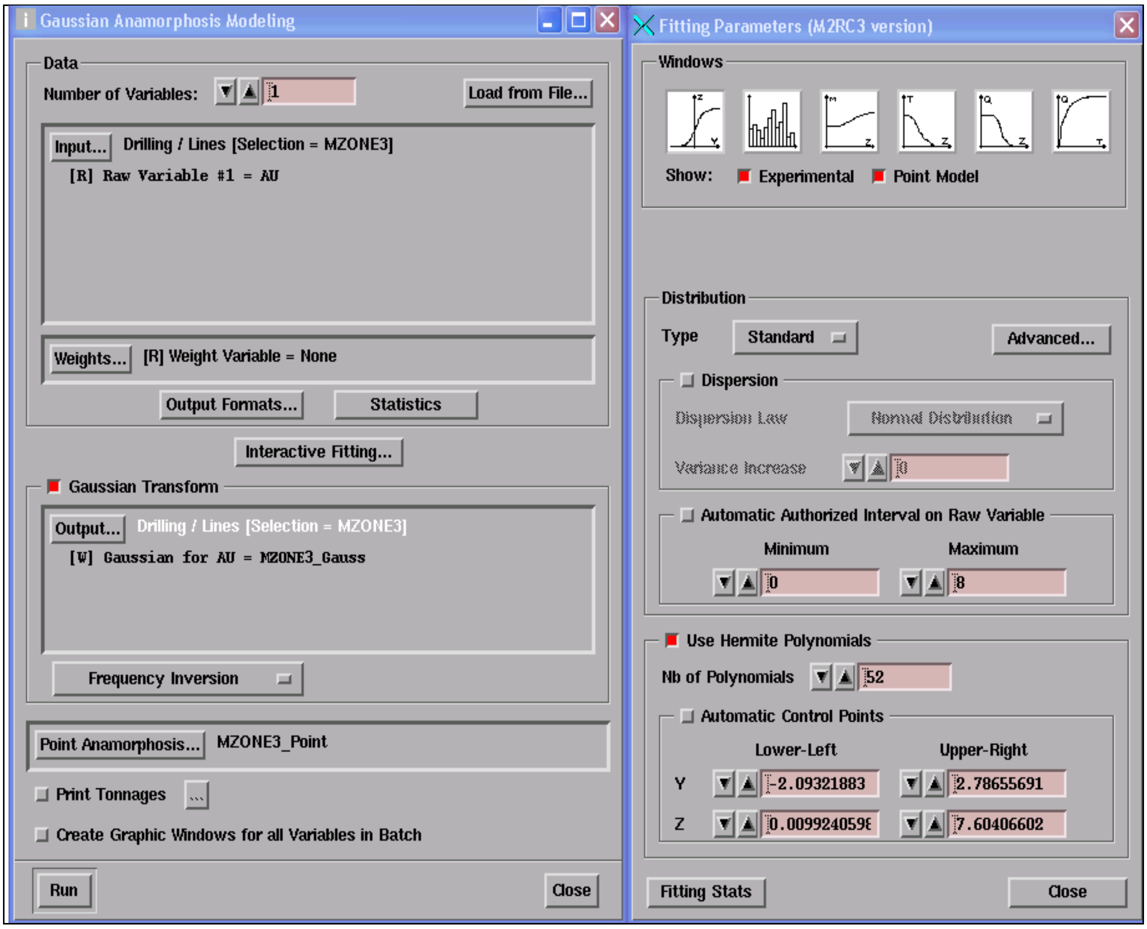
Data analysis for histogram window



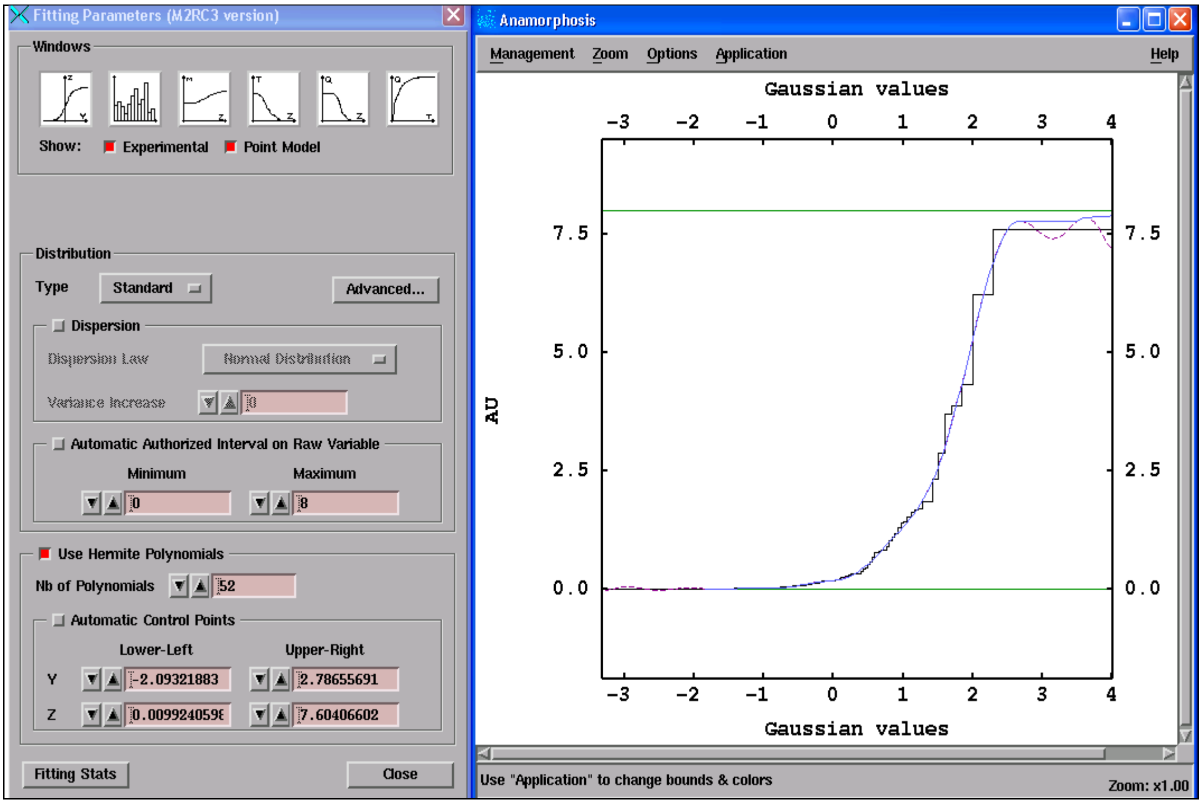
Exploratory data analysis-point anamorphosis

Exploratory data analysis is used to transform the raw gold (AU) values to a Gaussian distribution, because the simulation is based on the Gaussian model. To transform, run (*Statistics-->Gaussian anamorphosis modelling*). Save the transformed (Gaussian) value and the anamorphosis and check that the "Dispersion" is checked "OFF" in the "Interactive Fitting" panel.

Gaussian anamorphosis parameters window



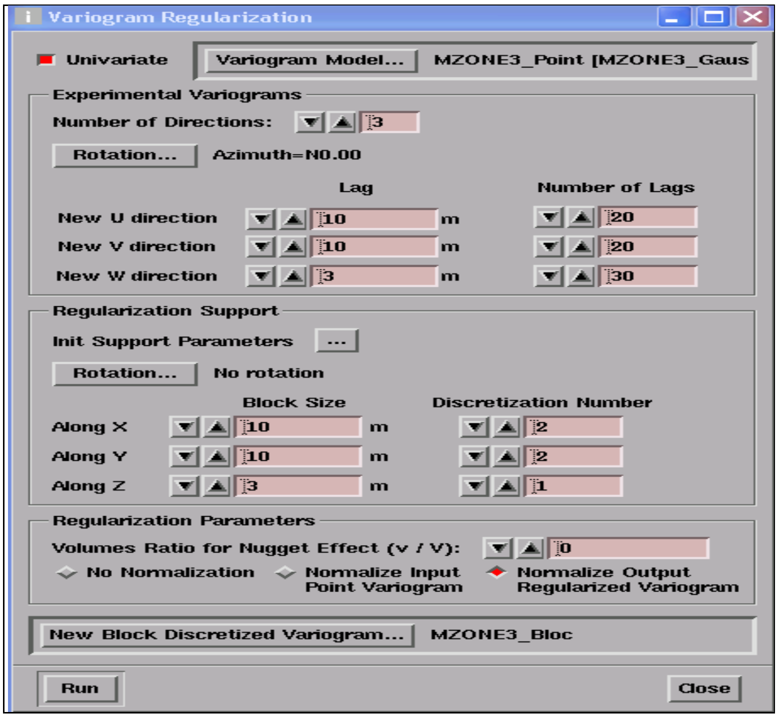
Interactive fitting parameters window



Variogram regularization

For the direct block simulation method, there is a need to regularise the point variogram model to a block variogram model. The path to launch is (Statistics->*Modelling->Variogram regularization*) Select variogram model, and gives name to new block discretisation. The lag size and number of lags in each direction needs to be sufficient to cover the ranges of the variograms, but still provide enough resolution. The block size is the dimensions of the SMU to be modelled. The volumes ratio for nugget effect negates the nugget effect on the assumption the nugget effect is very small relative to the size of the block. The choice of normalisation is up to the practitioner, but either option should have the same effect if the sample Gaussian variogram is modelled to a sill of 1, then store the regularised output variogram and fit the variogram model.

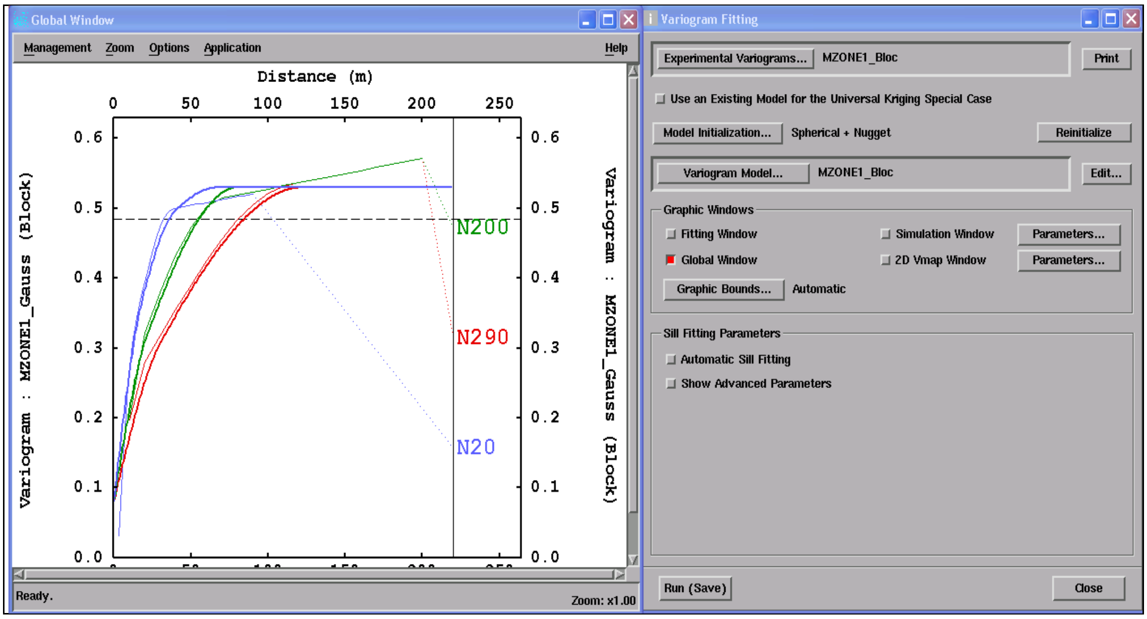
Variogram regularization window



Block variogram fitting

As with the point experimental variogram, the block variogram also needs to be fitted. The path is (*Statistics -> variogram fitting*), select the regularized variogram, and give a name to the variogram model.

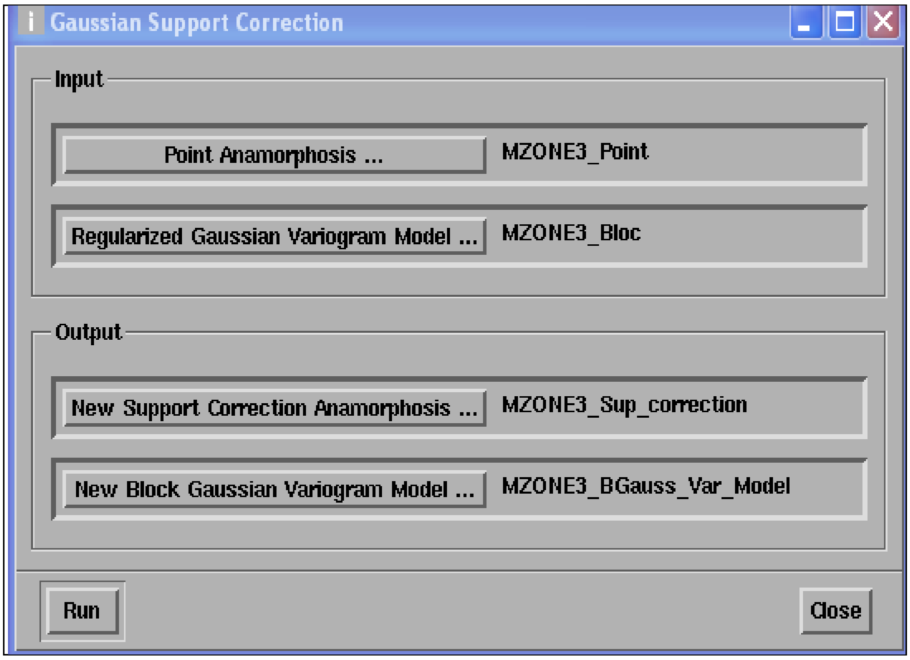
Block variogram fitting window



Block Gaussian support correction

The objective of this was to create a block anamorphosis that is similar to the point Gaussian anamorphosis. This is achieved by running (*Statistics->Modelling-->Gaussian support correction*). Select point Anamorphosis and Regularised Gaussian variable model; give a name to new support correction Anamorphosis and block Gauss variogram model. Support correction uses the point anamorphosis and the regularised Gaussian variogram to create a variogram model and change of support model in a format suitable for the DBSim.

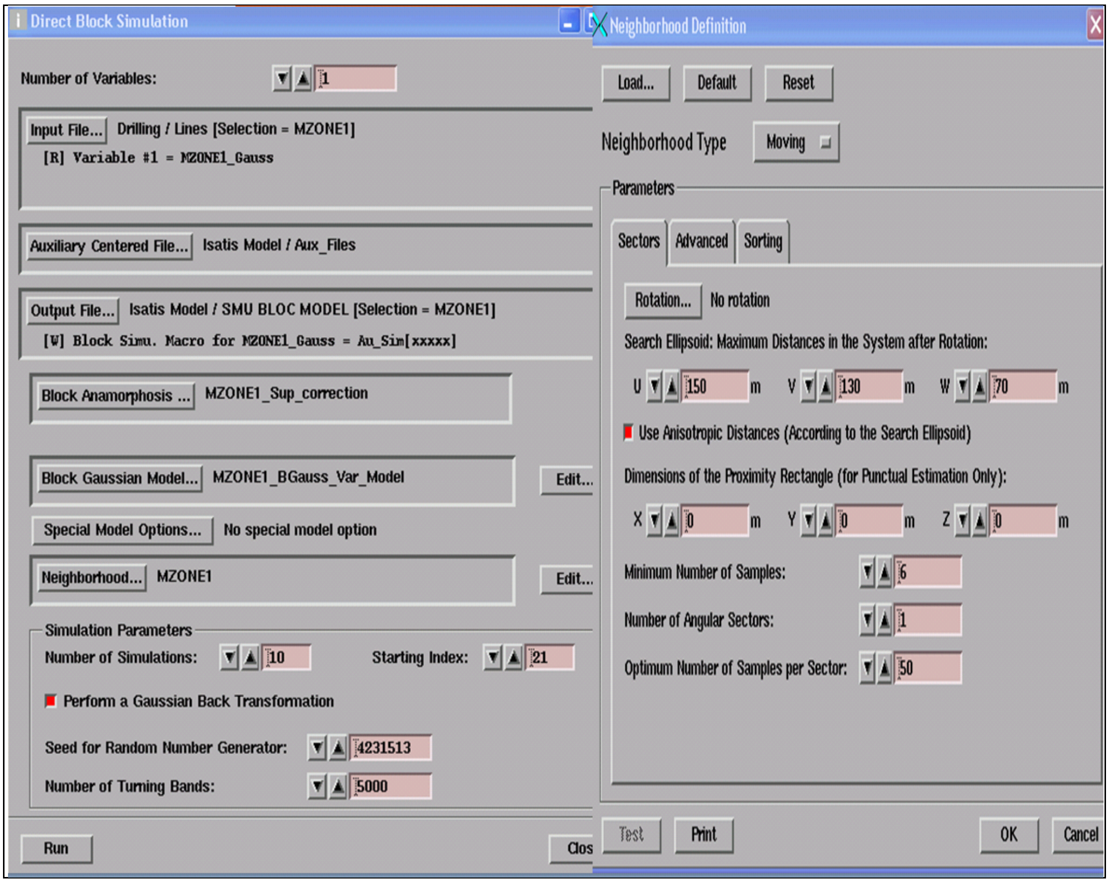
Block Gaussian support correction window



Direct Block simulation

The Direct Block simulation uses the Gaussian samples and stores the range of simulated values for each block in a macro variable. The path to launch is (*Interpolate -> conditional simulations -> direct block simulations*). Select Input file-Drilling lines, select zone, Aux file-, Block Anamorphosis-, Neighborhood.

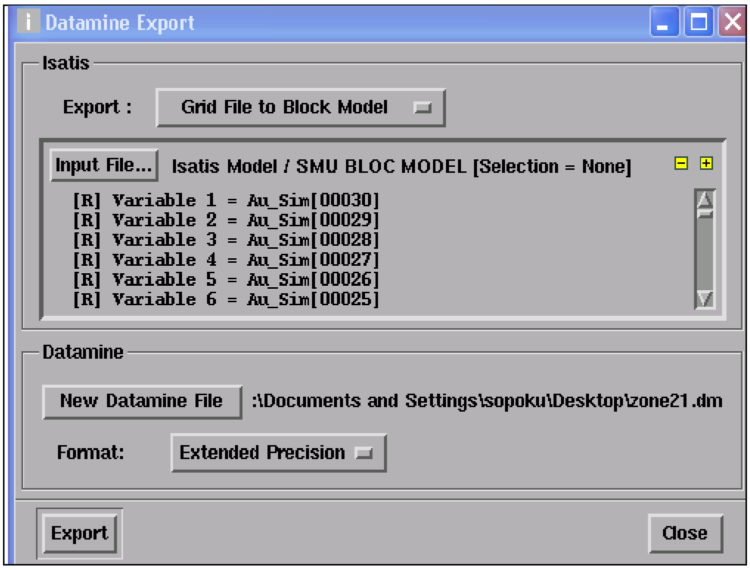
Direct block simulation neighborhood definition window



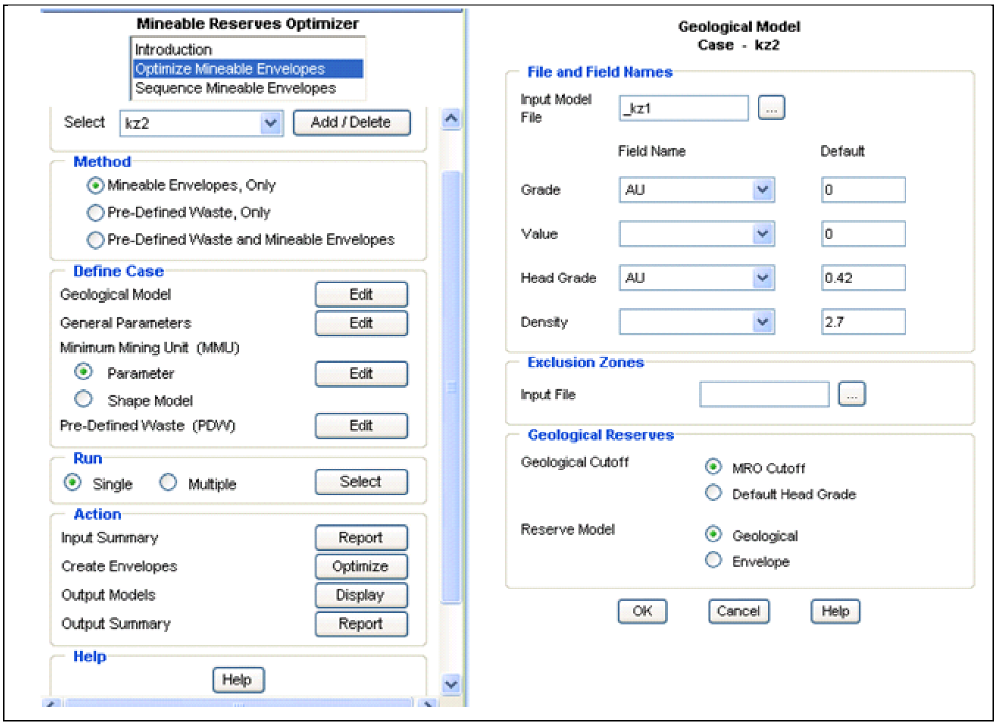
Exporting simulated model to Datamine format

The final simulated model is then exported to Datamine. The path to launch is (*File -> export ->Datamine*).

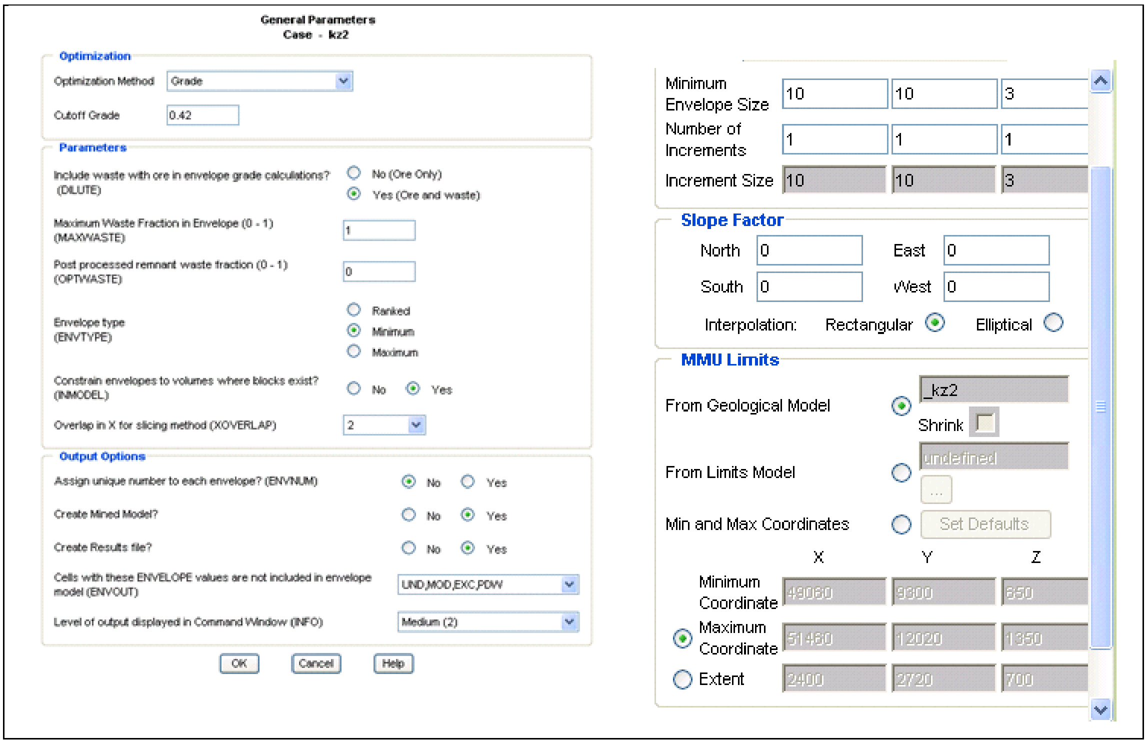
Exporting simulated models into Datamine window



Appendix 5: MRO Datamine script and input parameters



MRO key Input parameters



Appendix 6: Transition evaluation summary

Sadiola

				YEAR1	YEAR2	TOTAL		
1	Total Open Pit	tonnes		16,917,684	33,835,368	276,156,262		
2	Total Mining Cost	$		40,602,442	81,204,883	662,775,029		
3	Silver Tonnes Pit	tonnes		-	-	-		
4	Silver Content Pit	Ag_gm		-	-	-		
5	Silver Grade	Ag_g/t		-	-	-		
6	Au0 Tonnes Pit	tonnes		8,266,528	10,567,824	133,977,551		
7	Au0 Content Pit	Au_gm		17,142,596	18,760,994	296,943,920		
8	**Ore Tonnes**	**t**		8,266,528.02	10,567,824.35	133,977,551		
9	**Grade**	**g/t**		2.07	1.78	2.22		
10	**Waste Tonnes**	**t**		8,651,155.98	23,267,543.65	142,178,710.56		
11	**Strip Ratio**			**1.05**	**2.20**	**1.06**		
12								
13	Plant Feed Target	t		4,200,000	4,200,000	4,200,000		
14	Stockpile	t		8,266,528	14,634,352.37	133,977,551.44		
15		g/t		2.07	1.86	2.22		
16	Plant Feed	t		4,200,000	4,200,000	4,200,000		
17		g/t		2.07	1.86	2.22		
18								
19	Waste mining cost	US$	**2.40**	20,762,774	55,842,105	341,228,905		
20	Ore mining cost	US$	**2.40**	19,839,667	25,362,778	321,546,123		
21	Rehandle	US$	**0**	-	-	-		
22	Processing cost	US$	**33.951**	280,656,893	358,788,204	4,548,671,849		
23	G & A	US$	**0**	-	-	-		
24	Mine & Geol fixed	US$	**0**	-	-	-		
25	Mine closure	US$	**0**	-	-	-		
26	selling costs	US$	**2.99**	24,716,919	31,597,795	400,592,879		
27	Royalties	US$						
28								
29	**Total cost**			**345,976,253**	**471,590,882**	**5,612,039,757**		
30								
31	Revenue							
32	Tonnes	t		8,266,528	10,567,824	133,977,551		
33	Grade	g/t		2.07	1.78	2.22		
34	Total Gold	g		17,142,596	18,760,994	296,943,920		
35	Ounces per gram	oz/g		0.0321492	0.0321492	0.0321492		
36	Mining Ounces	oz		551,121	603,151	9,546,509		
37	Recovery	%		**0.93**	93%	93%		
38	Processed ounces	oz		512,542	560,930	8,878,254		
39	Gold price	US$/oz		**1300**	1300	1300		
40								
41	**Total Revenue**	US$		**666,304,982**	**729,209,496**	**11,541,729,945**		
42								
43	**Profit/Loss**	US$		**320,328,728**	**257,618,613**	**5,929,690,189**	NPV @ 10%	$3,053,540,250.20
44								
45	**Cost per Ounce**			**675**	**841**	**632**		
46	**Gold price/Cost per ounce**			1.93	1.55	2.06		

CVSA

				YEAR1	YEAR2	TOTAL		
1	Total Open Pit	tonnes		16,917,684	16,917,684	53,317,725		
2	Total Mining Cost	$		31,282,292	31,918,735	101,690,939		
3	Silver Tonnes Pit	tonnes		-	1,280	65,802		
4	Silver Content Pit	Ag_gm		-	200,689	12,777,933		
5	Silver Grade	Ag_g/t		-	157	552		
6	Au0 Tonnes Pit	tonnes		327,830	516,834	2,083,282		
7	Au0 Content Pit	Au_gm		2,852,598	4,926,868	19,662,846		
8	**Ore Tonnes**	**t**		327,829.84	516,834.15	2,083,282.12		
9	**Grade**	**g/t**		8.70	9.53	9.44		
10	**Waste Tonnes**	**t**		16,589,854.16	16,400,849.85	51,234,442.45		
11	**Strip Ratio**			**50.61**	**31.73**	**24.59**		
12								
13	Plant Feed Target	t		4,200,000	4,200,000	4,200,000		
14	Stockpile	t		327,830	516,834.15	2,083,282.12		
15		g/t		8.70	9.53	9.44		
16	Plant Feed	t		327,830	516,834	2,083,282		
17		g/t		8.70	9.53	9.44		
18								
19	Waste mining cost	US$	**1.86**	30,857,129	30,505,581	95,296,063		
20	Ore mining cost	US$	**1.86**	609,764	961,312	3,874,905		
21	Rehandle	US$	**0**	-	-	-		
22	Processing cost	US$	**51.358**	16,836,685	26,543,569	106,993,203		
23	G & A	US$	**0**	-	-	-		
24	Mine & Geol fixed	US$	**0**	-	-	-		
25	Mine closure	US$	**0**	-	-	-		
26	Selling costs	US$	**4.03**	1,321,154	2,082,842	8,395,627		
27	Royalties	US$						
28								
29	**Total cost**			**49,624,731**	**60,093,302**	**214,559,798**		
30								
31	Revenue							
32	Tonnes	t		327,830	516,834	2,083,282		
33	Grade	g/t		8.70	9.53	9.44		
34	Total Gold	g		2,852,598	4,926,868	19,662,846		
35	Ounces per gram	oz/g		0.0321492	0.0321492	0.0321492		
36	Mining Ounces	oz		91,709	158,395	632,145		
37	Recovery	%		**0.9429**	94%	94%		
38	Processed ounces	oz		86,472	149,351	596,049		
39	Gold price	US$/oz		1300	1300	1300		
40								
41	**Total Revenue**	US$		**112,413,822**	**194,155,681**	**774,864,101**		
42								
43	**Profit/Loss**	US$		**62,789,091**	**134,062,378**	**560,304,304**	NPV @ 10%	$437,622,204.25
44								
45	**Cost per Ounce**			**574**	**402**	**360**		
46	**Gold price/Cost per ounce**			2.27	3.23	3.61		

Morila

				YEAR1	YEAR2	TOTAL		
1	Total Open Pit	tonnes		31,859,298	11,220,442	185,763,330		
2	Total Mining Cost	$		146,818,128	51,669,363	855,318,926		
3	Silver Tonnes Pit	tonnes		-	-	-		
4	Silver Content Pit	Ag_gm		-	-	-		
5	Silver Grade	Ag_g/t		-	-	-		
6	Au0 Tonnes Pit	tonnes		4,200,000	4,200,000	39,161,400		
7	Au0 Content Pit	Au_gm		36,373,976	33,152,843	178,314,498		
8	**Ore Tonnes**	**t**		4,200,000.00	4,200,000.00	39,161,400.00		
9	**Grade**	**g/t**		8.66	7.89	4.55		
10	**Waste Tonnes**	**t**		27,659,298.10	7,020,441.90	146,601,930.00		
11	**Strip Ratio**			**6.59**	**1.67**	**3.74**		
12								
13	Plant Feed Target	t		4,200,000	4,200,000	4,200,000		
14	Stockpile	t		4,200,000	4,200,000.00	39,161,400.00		
15		g/t		8.66	7.89	4.55		
16	Plant Feed	t		4,200,000	4,200,000	4,200,000		
17		g/t		8.66	7.89	4.55		
18								
19	Waste mining cost	US$	**4.56**	126,126,399	32,013,215	668,504,801		
20	Ore mining cost	US$	**4.56**	19,152,000	19,152,000	178,575,984		
21	Rehandle	US$	**0**	-	-	-		
22	Processing cost	US$	**32.34**	135,828,000	135,828,000	1,266,479,676		
23	G & A	US$	**0**	-	-	-		
24	Mine & Geol fixed	US$	**0**	-	-	-		
25	Mine closure	US$	**0**	-	-	-		
26	Selling costs	US$	**2.96**	12,432,000	12,432,000	115,917,744		
27	Royalties	US$						
28								
29	**Total cost**			**293,538,399**	**199,425,215**	**2,229,478,205**		
30								
31	Revenue							
32	Tonnes	t		4,200,000	4,200,000	39,161,400		
33	Grade	g/t		8.66	7.89	4.55		
34	Total Gold	g		36,373,976	33,152,843	178,314,498		
35	Ounces per gram	oz/g		0.0321492	0.0321492	0.0321492		
36	Mining Ounces	oz		1,169,394	1,065,837	5,732,668		
37	Recovery	%		**0.89**	89%	89%		
38	Processed ounces	oz		1,040,761	948,595	5,102,075		
39	Gold price	US$/oz		1300	1300	1300		
40								
41	**Total Revenue**	US$		**1,352,989,133**	**1,233,173,852**	**6,632,697,408**		
42								
43	**Profit/Loss**	US$		**1,059,450,733**	**1,033,748,637**	**4,403,219,203**	NPV @ 10%	$3,413,598,177.21
44								
45	**Cost per Ounce**			**282**	**210**	**437**		
46	**Gold price/Cost per ounce**			4.61	6.18	2.97		

Geita

				YEAR1	YEAR2	TOTAL		
1	Total Open Pit	tonnes		16,917,684	33,835,369	390,154,601		
2	Total Mining Cost	$		19,193,762	36,516,185	488,917,959		
3	Silver Tonnes Pit	tonnes		-	-	-		
4	Silver Content Pit	Ag_gm		-	-	-		
5	Silver Grade	Ag_g/t		-	-	-		
6	Au0 Tonnes Pit	tonnes		3,890,864	2,025,445	46,268,588		
7	Au0 Content Pit	Au_gm		11,655,996	3,932,928	134,628,588		
8	**Ore Tonnes**	**t**		3,890,864.29	2,025,445.45	46,268,588		
9	**Grade**	**g/t**		3.00	1.94	2.91		
10	**Waste Tonnes**	**t**		13,026,820.01	31,809,923.15	343,886,012.81		
11	**Strip Ratio**			**3.35**	**15.71**	**7.43**		
12								
13	Plant Feed Target	t		4,200,000	4,200,000	4,200,000		
14	Stockpile	t		3,890,864	2,025,445.45	53,279,681.80		
15		g/t		3.00	1.94	2.95		
16	Plant Feed	t		3,890,864	2,025,445	4,200,000		
17		g/t		3.00	1.94	2.95		
18								
19	Waste mining cost	US$	**2.40**	31,264,368	76,343,816	825,326,431		
20	Ore mining cost	US$	**2.40**	9,338,074	4,861,069	111,044,612		
21	Rehandle	US$	**0**	-	-	-		
22	Processing cost	US$	**32.34**	125,830,551	65,502,906	1,496,326,142		
23	G & A	US$	**0**	-	-	-		
24	Mine & Geol fixed	US$	**0**	-	-	-		
25	Mine closure	US$	**0**	-	-	-		
26	Selling costs	US$	1.25	4,863,580	2,531,807	57,835,735		
27	Royalties	US$						
28								
29	**Total cost**			**171,296,574**	**149,239,597**	**2,490,532,919**		
30								
31	Revenue							
32	Tonnes	t		3,890,864	2,025,445	46,268,588		
33	Grade	g/t		3.00	1.94	2.91		
34	Total Gold	g		11,655,996	3,932,928	134,628,588		
35	Ounces per gram	oz/g		0.0321492	0.0321492	0.0321492		
36	Mining Ounces	oz		374,731	126,440	4,328,201		
37	Recovery	%		**0.89**	89%	89%		
38	Processed ounces	oz		333,511	112,532	3,852,099		
39	Gold price	US$/oz		**1300**	1300	1300		
40								
41	**Total Revenue**	US$		**433,563,714**	**146,291,643**	**5,007,729,034**		
42								
43	**Profit/Loss**	US$		**262,267,140**	**-2,947,954**	**2,517,196,115**	NPV @ 10%	$1,311,489,171.04
44								
45	**Cost per Ounce**			**514**	**1,326**	**647**		
46	**Gold price/Cost per ounce**			2.53	0.98	2.01		

Appendix 7: Macros used after optimisation

Macros for adding Whittle pit shells

```
Macro Name: add_OP.mac
!START 1

!LET $RUN#=0
!LET $RUNMAX#=100

!LOOP1:REM
!LET $RUN#=$RUN#+1
!IF $RUN# > $RUNMAX#, GOTO LOOPEND
!LET $R1#={int($RUN#)}

!LET $R2#=RUN$R1#

!FXIN     &OUT(_tem1),@FILETYPE=1.0
G:\WHT_XPAC\Results_Whittle\Geita\1300\OP\$R1#.par
G:\WHT_XPAC\Results_Whittle\Geita\1300\OP\$R1#.res

!COPY     &IN(_tem1),&OUT(_tem2),PIT=1

!MGSORT   &IN(_tem2),&OUT(_tem3),*KEY1(IJK),@ORDER=1.0

!EXTRA    &IN(_tem3),&OUT($R2#)

$R2#=1
ERASE(PIT)

GO

!OPSYS
del _tem1.dm _tem2.dm _tem3.dm

!GOTO LOOP1
!LOOPEND:REM

!ADDMOD   &IN1(run1),&IN2(run2),&OUT(_xx2),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx2),&IN2(run3),&OUT(_xx3),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx3),&IN2(run4),&OUT(_xx4),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx4),&IN2(run5),&OUT(_xx5),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx5),&IN2(run6),&OUT(_xx6),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx6),&IN2(run7),&OUT(_xx7),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx7),&IN2(run8),&OUT(_xx8),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx8),&IN2(run9),&OUT(_xx9),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx9),&IN2(run10),&OUT(_xx10),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx10),&IN2(run11),&OUT(_xx11),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx11),&IN2(run12),&OUT(_xx12),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx12),&IN2(run13),&OUT(_xx13),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx13),&IN2(run14),&OUT(_xx14),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx14),&IN2(run15),&OUT(_xx15),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx15),&IN2(run16),&OUT(_xx16),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx16),&IN2(run17),&OUT(_xx17),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx17),&IN2(run18),&OUT(_xx18),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx18),&IN2(run19),&OUT(_xx19),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx19),&IN2(run20),&OUT(_xx20),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx20),&IN2(run21),&OUT(_xx21),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx21),&IN2(run22),&OUT(_xx22),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx22),&IN2(run23),&OUT(_xx23),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx23),&IN2(run24),&OUT(_xx24),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx24),&IN2(run25),&OUT(_xx25),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx25),&IN2(run26),&OUT(_xx26),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx26),&IN2(run27),&OUT(_xx27),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx27),&IN2(run28),&OUT(_xx28),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx28),&IN2(run29),&OUT(_xx29),@TOLERNCE=0.001
```

```
!ADDMOD   &IN1(_xx29),&IN2(run30),&OUT(_xx30),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx30),&IN2(run31),&OUT(_xx31),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx31),&IN2(run32),&OUT(_xx32),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx32),&IN2(run33),&OUT(_xx33),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx33),&IN2(run34),&OUT(_xx34),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx34),&IN2(run35),&OUT(_xx35),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx35),&IN2(run36),&OUT(_xx36),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx36),&IN2(run37),&OUT(_xx37),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx37),&IN2(run38),&OUT(_xx38),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx38),&IN2(run39),&OUT(_xx39),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx39),&IN2(run40),&OUT(_xx40),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx40),&IN2(run41),&OUT(_xx41),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx41),&IN2(run42),&OUT(_xx42),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx42),&IN2(run43),&OUT(_xx43),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx43),&IN2(run44),&OUT(_xx44),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx44),&IN2(run45),&OUT(_xx45),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx45),&IN2(run46),&OUT(_xx46),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx46),&IN2(run47),&OUT(_xx47),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx47),&IN2(run48),&OUT(_xx48),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx48),&IN2(run49),&OUT(_xx49),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx49),&IN2(run50),&OUT(_xx50),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx50),&IN2(run51),&OUT(_xx51),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx51),&IN2(run52),&OUT(_xx52),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx52),&IN2(run53),&OUT(_xx53),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx53),&IN2(run54),&OUT(_xx54),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx54),&IN2(run55),&OUT(_xx55),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx55),&IN2(run56),&OUT(_xx56),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx56),&IN2(run57),&OUT(_xx57),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx57),&IN2(run58),&OUT(_xx58),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx58),&IN2(run59),&OUT(_xx59),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx59),&IN2(run60),&OUT(_xx60),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx60),&IN2(run61),&OUT(_xx61),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx61),&IN2(run62),&OUT(_xx62),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx62),&IN2(run63),&OUT(_xx63),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx63),&IN2(run64),&OUT(_xx64),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx64),&IN2(run65),&OUT(_xx65),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx65),&IN2(run66),&OUT(_xx66),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx66),&IN2(run67),&OUT(_xx67),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx67),&IN2(run68),&OUT(_xx68),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx68),&IN2(run69),&OUT(_xx69),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx69),&IN2(run70),&OUT(_xx70),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx70),&IN2(run71),&OUT(_xx71),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx71),&IN2(run72),&OUT(_xx72),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx72),&IN2(run73),&OUT(_xx73),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx73),&IN2(run74),&OUT(_xx74),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx74),&IN2(run75),&OUT(_xx75),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx75),&IN2(run76),&OUT(_xx76),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx76),&IN2(run77),&OUT(_xx77),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx77),&IN2(run78),&OUT(_xx78),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx78),&IN2(run79),&OUT(_xx79),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx79),&IN2(run80),&OUT(_xx80),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx80),&IN2(run81),&OUT(_xx81),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx81),&IN2(run82),&OUT(_xx82),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx82),&IN2(run83),&OUT(_xx83),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx83),&IN2(run84),&OUT(_xx84),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx84),&IN2(run85),&OUT(_xx85),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx85),&IN2(run86),&OUT(_xx86),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx86),&IN2(run87),&OUT(_xx87),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx87),&IN2(run88),&OUT(_xx88),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx88),&IN2(run89),&OUT(_xx89),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx89),&IN2(run90),&OUT(_xx90),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx90),&IN2(run91),&OUT(_xx91),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx91),&IN2(run92),&OUT(_xx92),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx92),&IN2(run93),&OUT(_xx93),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx93),&IN2(run94),&OUT(_xx94),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx94),&IN2(run95),&OUT(_xx95),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx95),&IN2(run96),&OUT(_xx96),@TOLERNCE=0.001
```

```
!ADDMOD   &IN1(_xx96),&IN2(run97),&OUT(_xx97),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx97),&IN2(run98),&OUT(_xx98),@TOLERNCE=0.001
!ADDMOD   &IN1(_xx98),&IN2(run99),&OUT(_xx99),@TOLERNCE=0.00
!ADDMOD   &IN1(_xx99),&IN2(run100),&OUT(_xx1),@TOLERNCE=0.001

!GENTRA &IN(_xx1),&OUT(T1)
EQC RUN1 -
SETC RUN1 0
EQC RUN2 -
SETC RUN2 0
EQC RUN3 -
SETC RUN3 0
EQC RUN4 -
SETC RUN4 0
EQC RUN5 -
SETC RUN5 0
EQC RUN6 -
SETC RUN6 0
EQC RUN7 -
SETC RUN7 0
EQC RUN8 -
SETC RUN8 0
EQC RUN9 -
SETC RUN9 0
EQC RUN10 -
SETC RUN10 0
EQC RUN11 -
SETC RUN11 0
EQC RUN12 -
SETC RUN12 0
EQC RUN13 -
SETC RUN13 0
EQC RUN14 -
SETC RUN14 0
EQC RUN15 -
SETC RUN15 0
EQC RUN16 -
SETC RUN16 0
EQC RUN17 -
SETC RUN17 0
EQC RUN18 -
SETC RUN18 0
EQC RUN19 -
SETC RUN19 0
EQC RUN20 -
SETC RUN20 0
EQC RUN21 -
SETC RUN21 0
EQC RUN22 -
SETC RUN22 0
EQC RUN23 -
SETC RUN23 0
EQC RUN24 -
SETC RUN24 0
EQC RUN25 -
SETC RUN25 0
EQC RUN26 -
SETC RUN26 0
EQC RUN27 -
SETC RUN27 0
EQC RUN28 -
SETC RUN28 0
EQC RUN29 -
SETC RUN29 0
EQC RUN30 -
SETC RUN30 0
EQC RUN31 -
```

SETC RUN31 0
EQC RUN32 -
SETC RUN32 0
EQC RUN33 -
SETC RUN33 0
EQC RUN34 -
SETC RUN34 0
EQC RUN35 -
SETC RUN35 0
EQC RUN36 -
SETC RUN36 0
EQC RUN37 -
SETC RUN37 0
EQC RUN38 -
SETC RUN38 0
EQC RUN39 -
SETC RUN39 0
EQC RUN40 -
SETC RUN40 0
EQC RUN41 -
SETC RUN41 0
EQC RUN42 -
SETC RUN42 0
EQC RUN43 -
SETC RUN43 0
EQC RUN44 -
SETC RUN44 0
EQC RUN45 -
SETC RUN45 0
EQC RUN46 -
SETC RUN46 0
EQC RUN47 -
SETC RUN47 0
EQC RUN48 -
SETC RUN48 0
EQC RUN49 -
SETC RUN49 0
EQC RUN50 -
SETC RUN50 0
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SETC RUN51 0
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SETC RUN52 0
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SETC RUN56 0
EQC RUN57 -
SETC RUN57 0
EQC RUN58 -
SETC RUN58 0
EQC RUN59 -
SETC RUN59 0
EQC RUN60 -
SETC RUN60 0
EQC RUN61 -
SETC RUN61 0
EQC RUN62 -
SETC RUN62 0
EQC RUN63 -
SETC RUN63 0
EQC RUN64 -
SETC RUN64 0

EQC RUN65 -
SETC RUN65 0
EQC RUN66 -
SETC RUN66 0
EQC RUN67 -
SETC RUN67 0
EQC RUN68 -
SETC RUN68 0
EQC RUN69 -
SETC RUN69 0
EQC RUN70 -
SETC RUN70 0
EQC RUN71 -
SETC RUN71 0
EQC RUN72 -
SETC RUN72 0
EQC RUN73 -
SETC RUN73 0
EQC RUN74 -
SETC RUN74 0
EQC RUN75 -
SETC RUN75 0
EQC RUN76 -
SETC RUN76 0
EQC RUN77 -
SETC RUN77 0
EQC RUN78 -
SETC RUN78 0
EQC RUN79 -
SETC RUN79 0
EQC RUN80 -
SETC RUN80 0
EQC RUN81 -
SETC RUN81 0
EQC RUN82 -
SETC RUN82 0
EQC RUN83 -
SETC RUN83 0
EQC RUN84 -
SETC RUN84 0
EQC RUN85 -
SETC RUN85 0
EQC RUN86 -
SETC RUN86 0
EQC RUN87 -
SETC RUN87 0
EQC RUN88 -
SETC RUN88 0
EQC RUN89 -
SETC RUN89 0
EQC RUN90 -
SETC RUN90 0
EQC RUN91 -
SETC RUN91 0
EQC RUN92 -
SETC RUN92 0
EQC RUN93 -
SETC RUN93 0
EQC RUN94 -
SETC RUN94 0
EQC RUN95 -
SETC RUN95 0
EQC RUN96 -
SETC RUN96 0
EQC RUN97 -
SETC RUN97 0
EQC RUN98 -

```
SETC RUN98 0
EQC RUN99 -
SETC RUN99 0
EQC RUN100 -
SETC RUN100 0
END
Y

!GENTRA &IN(T1),&OUT(OPMOD)
ADD T1 RUN1 RUN2
ADD T2 T1 RUN3
ADD T1 T2 RUN4
ADD T2 T1 RUN5
ADD T1 T2 RUN6
ADD T2 T1 RUN7
ADD T1 T2 RUN8
ADD T2 T1 RUN9
ADD T1 T2 RUN10
ADD T2 T1 RUN11
ADD T1 T2 RUN12
ADD T2 T1 RUN13
ADD T1 T2 RUN14
ADD T2 T1 RUN15
ADD T1 T2 RUN16
ADD T2 T1 RUN17
ADD T1 T2 RUN18
ADD T2 T1 RUN19
ADD T1 T2 RUN20
ADD T2 T1 RUN21
ADD T1 T2 RUN22
ADD T2 T1 RUN23
ADD T1 T2 RUN24
ADD T2 T1 RUN25
ADD T1 T2 RUN26
ADD T2 T1 RUN27
ADD T1 T2 RUN28
ADD T2 T1 RUN29
ADD T1 T2 RUN30
ADD T2 T1 RUN31
ADD T1 T2 RUN32
ADD T2 T1 RUN33
ADD T1 T2 RUN34
ADD T2 T1 RUN35
ADD T1 T2 RUN36
ADD T2 T1 RUN37
ADD T1 T2 RUN38
ADD T2 T1 RUN39
ADD T1 T2 RUN40
ADD T2 T1 RUN41
ADD T1 T2 RUN42
ADD T2 T1 RUN43
ADD T1 T2 RUN44
ADD T2 T1 RUN45
ADD T1 T2 RUN46
ADD T2 T1 RUN47
ADD T1 T2 RUN48
ADD T2 T1 RUN49
ADD T1 T2 RUN50
ADD T2 T1 RUN51
ADD T1 T2 RUN52
ADD T2 T1 RUN53
ADD T1 T2 RUN54
ADD T2 T1 RUN55
ADD T1 T2 RUN56
ADD T2 T1 RUN57
ADD T1 T2 RUN58
ADD T2 T1 RUN59
```

```
ADD T1 T2 RUN60
ADD T2 T1 RUN61
ADD T1 T2 RUN62
ADD T2 T1 RUN63
ADD T1 T2 RUN64
ADD T2 T1 RUN65
ADD T1 T2 RUN66
ADD T2 T1 RUN67
ADD T1 T2 RUN68
ADD T2 T1 RUN69
ADD T1 T2 RUN70
ADD T2 T1 RUN71
ADD T1 T2 RUN72
ADD T2 T1 RUN73
ADD T1 T2 RUN74
ADD T2 T1 RUN75
ADD T1 T2 RUN76
ADD T2 T1 RUN77
ADD T1 T2 RUN78
ADD T2 T1 RUN79
ADD T1 T2 RUN80
ADD T2 T1 RUN81
ADD T1 T2 RUN82
ADD T2 T1 RUN83
ADD T1 T2 RUN84
ADD T2 T1 RUN85
ADD T1 T2 RUN86
ADD T2 T1 RUN87
ADD T1 T2 RUN88
ADD T2 T1 RUN89
ADD T1 T2 RUN90
ADD T2 T1 RUN91
ADD T1 T2 RUN92
ADD T2 T1 RUN93
ADD T1 T2 RUN94
ADD T2 T1 RUN95
ADD T1 T2 RUN96
ADD T2 T1 RUN97
ADD T1 T2 RUN98
ADD T2 T1 RUN99
ADD T1 T2 RUN100
DIVC PB T1 100
ERA  T2;
END
Y

!DELETE &IN(_xx1),@Confirm=0.0
!DELETE &IN(_xx2),@Confirm=0.0
!DELETE &IN(_xx3),@Confirm=0.0
!DELETE &IN(_xx4),@Confirm=0.0
!DELETE &IN(_xx5),@Confirm=0.0
!DELETE &IN(_xx6),@Confirm=0.0
!DELETE &IN(_xx7),@Confirm=0.0
!DELETE &IN(_xx8),@Confirm=0.0
!DELETE &IN(_xx9),@Confirm=0.0
!DELETE &IN(_xx10),@Confirm=0.0
!DELETE &IN(_xx11),@Confirm=0.0
!DELETE &IN(_xx12),@Confirm=0.0
!DELETE &IN(_xx13),@Confirm=0.0
!DELETE &IN(_xx14),@Confirm=0.0
!DELETE &IN(_xx15),@Confirm=0.0
!DELETE &IN(_xx16),@Confirm=0.0
!DELETE &IN(_xx17),@Confirm=0.0
!DELETE &IN(_xx18),@Confirm=0.0
!DELETE &IN(_xx18),@Confirm=0.0
!DELETE &IN(_xx20),@Confirm=0.0
!DELETE &IN(_xx21),@Confirm=0.0
```

```
!DELETE &IN(_xx22),@Confirm=0.0
!DELETE &IN(_xx23),@Confirm=0.0
!DELETE &IN(_xx24),@Confirm=0.0
!DELETE &IN(_xx25),@Confirm=0.0
!DELETE &IN(_xx26),@Confirm=0.0
!DELETE &IN(_xx27),@Confirm=0.0
!DELETE &IN(_xx28),@Confirm=0.0
!DELETE &IN(_xx29),@Confirm=0.0
!DELETE &IN(_xx30),@Confirm=0.0
!DELETE &IN(_xx31),@Confirm=0.0
!DELETE &IN(_xx32),@Confirm=0.0
!DELETE &IN(_xx33),@Confirm=0.0
!DELETE &IN(_xx34),@Confirm=0.0
!DELETE &IN(_xx35),@Confirm=0.0
!DELETE &IN(_xx36),@Confirm=0.0
!DELETE &IN(_xx37),@Confirm=0.0
!DELETE &IN(_xx38),@Confirm=0.0
!DELETE &IN(_xx39),@Confirm=0.0
!DELETE &IN(_xx40),@Confirm=0.0
!DELETE &IN(_xx41),@Confirm=0.0
!DELETE &IN(_xx42),@Confirm=0.0
!DELETE &IN(_xx43),@Confirm=0.0
!DELETE &IN(_xx44),@Confirm=0.0
!DELETE &IN(_xx45),@Confirm=0.0
!DELETE &IN(_xx46),@Confirm=0.0
!DELETE &IN(_xx47),@Confirm=0.0
!DELETE &IN(_xx48),@Confirm=0.0
!DELETE &IN(_xx49),@Confirm=0.0
!DELETE &IN(_xx50),@Confirm=0.0
!DELETE &IN(_xx51),@Confirm=0.0
!DELETE &IN(_xx52),@Confirm=0.0
!DELETE &IN(_xx53),@Confirm=0.0
!DELETE &IN(_xx54),@Confirm=0.0
!DELETE &IN(_xx55),@Confirm=0.0
!DELETE &IN(_xx56),@Confirm=0.0
!DELETE &IN(_xx57),@Confirm=0.0
!DELETE &IN(_xx58),@Confirm=0.0
!DELETE &IN(_xx59),@Confirm=0.0
!DELETE &IN(_xx60),@Confirm=0.0
!DELETE &IN(_xx61),@Confirm=0.0
!DELETE &IN(_xx62),@Confirm=0.0
!DELETE &IN(_xx63),@Confirm=0.0
!DELETE &IN(_xx64),@Confirm=0.0
!DELETE &IN(_xx65),@Confirm=0.0
!DELETE &IN(_xx66),@Confirm=0.0
!DELETE &IN(_xx67),@Confirm=0.0
!DELETE &IN(_xx68),@Confirm=0.0
!DELETE &IN(_xx69),@Confirm=0.0
!DELETE &IN(_xx70),@Confirm=0.0
!DELETE &IN(_xx71),@Confirm=0.0
!DELETE &IN(_xx72),@Confirm=0.0
!DELETE &IN(_xx73),@Confirm=0.0
!DELETE &IN(_xx74),@Confirm=0.0
!DELETE &IN(_xx75),@Confirm=0.0
!DELETE &IN(_xx76),@Confirm=0.0
!DELETE &IN(_xx77),@Confirm=0.0
!DELETE &IN(_xx78),@Confirm=0.0
!DELETE &IN(_xx79),@Confirm=0.0
!DELETE &IN(_xx80),@Confirm=0.0
!DELETE &IN(_xx81),@Confirm=0.0
!DELETE &IN(_xx82),@Confirm=0.0
!DELETE &IN(_xx83),@Confirm=0.0
!DELETE &IN(_xx84),@Confirm=0.0
!DELETE &IN(_xx85),@Confirm=0.0
!DELETE &IN(_xx86),@Confirm=0.0
!DELETE &IN(_xx87),@Confirm=0.0
!DELETE &IN(_xx88),@Confirm=0.0
```

```
!DELETE &IN(_xx89),@Confirm=0.0
!DELETE &IN(_xx90),@Confirm=0.0
!DELETE &IN(_xx91),@Confirm=0.0
!DELETE &IN(_xx92),@Confirm=0.0
!DELETE &IN(_xx93),@Confirm=0.0
!DELETE &IN(_xx94),@Confirm=0.0
!DELETE &IN(_xx95),@Confirm=0.0
!DELETE &IN(_xx96),@Confirm=0.0
!DELETE &IN(_xx97),@Confirm=0.0
!DELETE &IN(_xx98),@Confirm=0.0
!DELETE &IN(_xx99),@Confirm=0.0

!DELETE &IN(RUN1),@Confirm=0.0
!DELETE &IN(RUN2),@Confirm=0.0
!DELETE &IN(RUN3),@Confirm=0.0
!DELETE &IN(RUN4),@Confirm=0.0
!DELETE &IN(RUN5),@Confirm=0.0
!DELETE &IN(RUN6),@Confirm=0.0
!DELETE &IN(RUN7),@Confirm=0.0
!DELETE &IN(RUN8),@Confirm=0.0
!DELETE &IN(RUN9),@Confirm=0.0
!DELETE &IN(RUN10),@Confirm=0.0
!DELETE &IN(RUN11),@Confirm=0.0
!DELETE &IN(RUN12),@Confirm=0.0
!DELETE &IN(RUN13),@Confirm=0.0
!DELETE &IN(RUN14),@Confirm=0.0
!DELETE &IN(RUN15),@Confirm=0.0
!DELETE &IN(RUN16),@Confirm=0.0
!DELETE &IN(RUN17),@Confirm=0.0
!DELETE &IN(RUN18),@Confirm=0.0
!DELETE &IN(RUN19),@Confirm=0.0
!DELETE &IN(RUN20),@Confirm=0.0
!DELETE &IN(RUN21),@Confirm=0.0
!DELETE &IN(RUN22),@Confirm=0.0
!DELETE &IN(RUN23),@Confirm=0.0
!DELETE &IN(RUN24),@Confirm=0.0
!DELETE &IN(RUN25),@Confirm=0.0
!DELETE &IN(RUN26),@Confirm=0.0
!DELETE &IN(RUN27),@Confirm=0.0
!DELETE &IN(RUN28),@Confirm=0.0
!DELETE &IN(RUN29),@Confirm=0.0
!DELETE &IN(RUN30),@Confirm=0.0
!DELETE &IN(RUN31),@Confirm=0.0
!DELETE &IN(RUN32),@Confirm=0.0
!DELETE &IN(RUN33),@Confirm=0.0
!DELETE &IN(RUN34),@Confirm=0.0
!DELETE &IN(RUN35),@Confirm=0.0
!DELETE &IN(RUN36),@Confirm=0.0
!DELETE &IN(RUN37),@Confirm=0.0
!DELETE &IN(RUN38),@Confirm=0.0
!DELETE &IN(RUN39),@Confirm=0.0
!DELETE &IN(RUN40),@Confirm=0.0
!DELETE &IN(RUN41),@Confirm=0.0
!DELETE &IN(RUN42),@Confirm=0.0
!DELETE &IN(RUN43),@Confirm=0.0
!DELETE &IN(RUN44),@Confirm=0.0
!DELETE &IN(RUN45),@Confirm=0.0
!DELETE &IN(RUN46),@Confirm=0.0
!DELETE &IN(RUN47),@Confirm=0.0
!DELETE &IN(RUN48),@Confirm=0.0
!DELETE &IN(RUN49),@Confirm=0.0
!DELETE &IN(RUN50),@Confirm=0.0
!DELETE &IN(RUN51),@Confirm=0.0
!DELETE &IN(RUN52),@Confirm=0.0
!DELETE &IN(RUN53),@Confirm=0.0
!DELETE &IN(RUN54),@Confirm=0.0
!DELETE &IN(RUN55),@Confirm=0.0
```

```
!DELETE &IN(RUN56),@Confirm=0.0
!DELETE &IN(RUN57),@Confirm=0.0
!DELETE &IN(RUN58),@Confirm=0.0
!DELETE &IN(RUN59),@Confirm=0.0
!DELETE &IN(RUN60),@Confirm=0.0
!DELETE &IN(RUN61),@Confirm=0.0
!DELETE &IN(RUN62),@Confirm=0.0
!DELETE &IN(RUN63),@Confirm=0.0
!DELETE &IN(RUN64),@Confirm=0.0
!DELETE &IN(RUN65),@Confirm=0.0
!DELETE &IN(RUN66),@Confirm=0.0
!DELETE &IN(RUN67),@Confirm=0.0
!DELETE &IN(RUN68),@Confirm=0.0
!DELETE &IN(RUN69),@Confirm=0.0
!DELETE &IN(RUN70),@Confirm=0.0
!DELETE &IN(RUN71),@Confirm=0.0
!DELETE &IN(RUN72),@Confirm=0.0
!DELETE &IN(RUN73),@Confirm=0.0
!DELETE &IN(RUN74),@Confirm=0.0
!DELETE &IN(RUN75),@Confirm=0.0
!DELETE &IN(RUN76),@Confirm=0.0
!DELETE &IN(RUN77),@Confirm=0.0
!DELETE &IN(RUN78),@Confirm=0.0
!DELETE &IN(RUN79),@Confirm=0.0
!DELETE &IN(RUN80),@Confirm=0.0
!DELETE &IN(RUN81),@Confirm=0.0
!DELETE &IN(RUN82),@Confirm=0.0
!DELETE &IN(RUN83),@Confirm=0.0
!DELETE &IN(RUN84),@Confirm=0.0
!DELETE &IN(RUN85),@Confirm=0.0
!DELETE &IN(RUN86),@Confirm=0.0
!DELETE &IN(RUN87),@Confirm=0.0
!DELETE &IN(RUN88),@Confirm=0.0
!DELETE &IN(RUN89),@Confirm=0.0
!DELETE &IN(RUN90),@Confirm=0.0
!DELETE &IN(RUN91),@Confirm=0.0
!DELETE &IN(RUN92),@Confirm=0.0
!DELETE &IN(RUN93),@Confirm=0.0
!DELETE &IN(RUN94),@Confirm=0.0
!DELETE &IN(RUN95),@Confirm=0.0
!DELETE &IN(RUN96),@Confirm=0.0
!DELETE &IN(RUN97),@Confirm=0.0
!DELETE &IN(RUN98),@Confirm=0.0
!DELETE &IN(RUN99),@Confirm=0.0
!DELETE &IN(RUN100),@Confirm=0.0

!END
```

Macros for converting block models to wireframes

Macro Name: BM2WF3.mac

```
!START B000      Lerchs Grossmann plots

------------------------------------------------------------
  Use the regularised blocks from the LG pit where exist
  & fill in rest of surface from the original model

------------------------------------------------------------
------------------------------------------------------------
  DEFINE INPUT MODEL, lg model & output points file NUMBER (run no.)
  ORIGINS & EXTENT OF LG MODEL MUST BE THOSE OF THE INPUT MODEL.
--------------------------------------------------------------------
   Just get LG model values

!PROMPT
```

```
0
0
1   Enter the name of the LG Model ............. : '$MODEL#',A,8
0
!COPY &IN($MODEL#),&OUT(TEMP)
!field $exist#=$MODEL#,$recs#=0,$LGNX=NX,$LGNY=NY,$LGNZ=NZ,$ZMORIG=ZMORIG
!field $exist#=$MODEL#,$recs#=0,$LGXINC=XINC,$LGYINC=YINC,$LGZINC=ZINC
!field $exist#=$MODEL#,$recs#=0,$XMORIG=XMORIG,$YMORIG=YMORIG
!let $ZRANGE=$LGNZ*$LGZINC
!let $ZRANGE=$ZRANGE

select points at the base of the regular LG model

!PROTOM &OUT(TPROT)
N
Y
$XMORIG
$YMORIG
$ZMORIG
$LGXINC
$LGYINC
$ZRANGE
$LGNX
$LGNY
1

!EDIT &IN(TEMP)
V
ZINC
$ZRANGE
V
NZ
1
E
!IJKGEN &PROTO(TPROT),&IN(TEMP),&OUT(TEMP),*X(XC),*Y(YC),
        *Z(ZC),@PSMODEL=0
!SORT &IN(TEMP),&OUT(TEMP1),*KEY1(IJK),*KEY2(ZC),IJK>-1
!VALIDA &IN(TEMP1),&OUT(TEMP2)
TEST IJK     .NE.IJK
LAST
#!GENTRA &IN(TEMP2),&OUT(TEMP3A)
DIVC HALFZINC ZINC 2
SUB Z ZC HALFZINC
THIS X XC
THIS Y YC
END
OK
#

!GENTRA &IN(TEMP2),&OUT(TEMP3A)
DIVC HALFZINC ZINC 2
THIS Z ZC
THIS X XC
THIS Y YC
END
OK
#

!SURTRI &WIREPT(zone1pt),&WIRETR(zone1tr),
       &POINTIN(temp3a),*XPT(X),*YPT(Y),*ZPT(Z),@COG=0,
       @SURFACE=1.0,@SYSTEM=3.0,@ERRTRACE=1,@MAXLINK=80

!END
```

Macros for creating pushbacks

Macro Name: add_pb.mac

```
!START 1


!FXIN     &OUT(_temp1),@FILETYPE=1.0
G:\WHT_XPAC\Geita\GEITA DATAMINE\2012 Geita\pb.par
G:\WHT_XPAC\Geita\GEITA DATAMINE\2012 Geita\pb.res


!MGSORT   &IN(_temp1),&OUT(_temp2),*KEY1(IJK),@ORDER=1.0

!EXTRA    &IN(_temp2),&OUT(PITX1)

IF (PIT<=6) PIT=6 end
IF (PIT<=13 & PIT>6) PIT=13 end
IF (PIT<=15 & PIT>13) PIT=15 end
IF (PIT<=22 & PIT>15) PIT=22 end
IF (PIT<=28 & PIT>22) PIT=28 end
IF (PIT<=36 & PIT>28) PIT=36 end

GO

!END
```

Macros to create input models for evaluation (XPAC)

Macro Name: pp.mac

```
!START 1

!SELWF&IN(ausimm1),&WIRETR(ugtr),&WIREPT(ugpt),&OUT(_ss1),*X(XC),*Y(YC),*Z(ZC),@SELECT=1
.0,@EXCLUDE=0.0,@TOLERANC=0.001

!SELWF
&IN(ausimm1),&WIRETR(optr),&WIREPT(oppt),&OUT(_ss2),*X(XC),*Y(YC),*Z(ZC),@SELECT=1.0,@EX
CLUDE=0.0, @TOLERANC=0.001

!SELWF&IN(ausimm1),&WIRETR(ugtr),&WIREPT(ugpt),&OUT(_ss1m),*X(XC),*Y(YC),*Z(ZC),@SELECT
=1.0,@EXCLUDE=0.0,@TOLERANC=0.001

!SELWF&IN(ausimm1),&WIRETR(optr),&WIREPT(oppt),&OUT(_ss2m),*X(XC),*Y(YC),*Z(ZC),@SELECT
=1.0,@EXCLUDE=0.0,@TOLERANC=0.001

!SELWF&IN(ausimm1),&WIRETR(ugtr),&WIREPT(ugpt),&OUT(_s2),*X(XC),*Y(YC),*Z(ZC),@SELECT=2.
0,@EXCLUDE=0.0, @TOLERANC=0.001

!SELWF&IN(ausimm1),&WIRETR(optr),&WIREPT(oppt),&OUT(_ss3m),*X(XC),*Y(YC),*Z(ZC),@SELECT
=1.0,@EXCLUDE=0.0,@TOLERANC=0.001

!SELWF&IN(ausimm1),&WIRETR(ugtr),&WIREPT(ugpt),OUT(_ss4m),*X(XC),*Y(YC),*Z(ZC),@SELECT=2
.0,@EXCLUDE=0.0,
        @TOLERANC=0.001


!PROTOM   &OUT(prot),@ROTMOD=0.0
n
n
49060
9300
650
10
10
10
```

420
272
70

*!REGMOD &IN1(prot),&IN2(ugenv),&OUT(ugenv1),*F1(ENVNUM),*F2(DENSITY),*F2(AU)*

!ADDMOD &IN1(ugenv1),&IN2(_ss2m),&OUT(_xx2),@TOLERNCE=0.001

!COPY &IN(_xx2),&OUT(_ss4m),ENVNUM=1.0

*!SELWF&IN(_ss4m),&WIRETR(ugtr),&WIREPT(ugpt),&OUT(_ss3m),*X(XC),*Y(YC),*Z(ZC),@SELECT=2.0,@EXCLUDE=0.0,@TOLERANC=0.001*

!END

XPAC preparation macros

Macro Name: OPdm2XPAC_ss_all2.mac

```
!start 1
!let $folder = 'G:\WHT_XPAC\Geita\GEITA DATAMINE\XPAC'

!let $IN# = '_ss2m'
!let $Option = 'OP'
!let $Op# = 1
!let $PITOFF = 0.90

!rem Reduce X& Y coordinate digits for XPAC
!let $EAST = 0
!let $NORTH = 0
!let $LEVEL= 0

!field $EXIST#=$IN#,$recs#=0,$XORIG#=XMORIG,$YORIG#=YMORIG,$ZORIG#=ZMORIG
!field $EXIST#=$IN#,$recs#=0,$NX#=NX,$NY#=NY,$NZ#=NZ
!field $EXIST#=$IN#,$recs#=0,$XINC#=XINC,$YINC#=YINC,$ZINC#=ZINC

!rem - slice models  10m BENCH

!let $SXINC#=10
!let $SYINC#=10
!let $SZINC#=10
!let $01#={int($NX#/($SXINC#/$XINC#))+1}
!let $02#={int($NY#/($SYINC#/$YINC#))+1}
!let $03#={int($NZ#/($SZINC#/$ZINC#))+1}

!PROTOM &OUT(TPROT)
N
Y
$XORIG#
$YORIG#
$ZORIG#
$SXINC#
$SYINC#
$SZINC#
$01#
$02#
$03#

!LET $Pa = 1
!LET $Pb = 2
!LET $Pc = 3
!LET $Pd = 4
!LET $Pe = 5
!LET $Pf = 6
!LET $Pg = 7
```

```
!LET $Ph = 8
!LET $Pi = 9
!LET $Pj = 10
!LET $RUN =  'run1'
!LET $RETURN#=S1
!GOTO SUB1

****** S1 **********
!S1:REM
!LET $Pa =11
!LET $Pb =12
!LET $Pc =13
!LET $Pd =14
!LET $Pe =15
!LET $Pf =16
!LET $Pg =17
!LET $Ph =18
!LET $Pi =19
!LET $Pj =20
!LET $RUN =  'run2'
!LET $RETURN#=S2
!GOTO SUB1
****** S2 **********
!S2:REM
!LET $Pa =21
!LET $Pb =22
!LET $Pc =23
!LET $Pd =24
!LET $Pe =25
!LET $Pf =26
!LET $Pg =27
!LET $Ph =28
!LET $Pi =29
!LET $Pj =30
!LET $RUN =  'run3'
!LET $RETURN#=S3
!GOTO SUB1

****** S3 **********
!S3:REM

!LET $Pa =31
!LET $Pb =32
!LET $Pc =33
!LET $Pd =34
!LET $Pe =35
!LET $Pf =36
!LET $Pg =37
!LET $Ph =38
!LET $Pi =39
!LET $Pj =40
!LET $RUN =  'run4'
!LET $RETURN#=S4
!GOTO SUB1

****** S4 **********
!S4:REM

!LET $Pa =41
!LET $Pb =42
!LET $Pc =43
!LET $Pd =44
!LET $Pe =45
!LET $Pf =46
!LET $Pg =47
!LET $Ph =48
!LET $Pi =49
```

```
!LET $Pj =50
!LET $RUN =  'run5'
!LET $RETURN#=S5
!GOTO SUB1

****** S5 **********
!S5:REM

!LET $Pa =51
!LET $Pb =52
!LET $Pc =53
!LET $Pd =54
!LET $Pe =55
!LET $Pf =56
!LET $Pg =57
!LET $Ph =58
!LET $Pi =59
!LET $Pj =60
!LET $RUN =  'run6'
!LET $RETURN#=S6
!GOTO SUB1

****** S6 **********
!S6:REM

!LET $Pa =61
!LET $Pb =62
!LET $Pc =63
!LET $Pd =64
!LET $Pe =65
!LET $Pf =66
!LET $Pg =67
!LET $Ph =68
!LET $Pi =69
!LET $Pj =70
!LET $RUN =  'run7'
!LET $RETURN#=S7
!GOTO SUB1

****** S7 **********
!S7:REM

!LET $Pa =71
!LET $Pb =72
!LET $Pc =73
!LET $Pd =74
!LET $Pe =75
!LET $Pf =76
!LET $Pg =77
!LET $Ph =78
!LET $Pi =79
!LET $Pj =80
!LET $RUN =  'run8'
!LET $RETURN#=S8
!GOTO SUB1

****** S8 **********
!S8:REM

!LET $Pa =81
!LET $Pb =82
!LET $Pc =83
!LET $Pd =84
!LET $Pe =85
!LET $Pf =86
!LET $Pg =87
!LET $Ph =88
```

```
!LET $Pi =89
!LET $Pj =90
!LET $RUN =  'run9'
!LET $RETURN#=S9
!GOTO SUB1
****** S9 **********
!S9:REM

!LET $Pa =91
!LET $Pb =92
!LET $Pc =93
!LET $Pd =94
!LET $Pe =95
!LET $Pf =96
!LET $Pg =97
!LET $Ph =98
!LET $Pi =99
!LET $Pj =100
!LET $RUN =  'run10'
!LET $RETURN#=FINISH
!GOTO SUB1

!SUB1:REM

!INPFIL   &OUT(_field)

FIELDNAM
A
8
Y

$
Y

IJK
DENSITY
PIT
AU
$Pa
$Pb
$Pc
$Pd
$Pe
$Pf
$Pg
$Ph
$Pi
$Pj
XC
YC
ZC
XMORIG
YMORIG
ZMORIG
NX
NY
NZ
XINC
YINC
ZINC
!
!SELCOPY   &IN($IN#),&OUT(T1),&FIELDLST(_field),*F1(IJK),
           @KEEPALL=1.0

!COPY     &IN(T1),&OUT(T4),PIT>0.1<+

!EXTRA    &IN(T4),&OUT(T5)
```

```
AU$Pa = $Pa
AU$Pb = $Pb
AU$Pc = $Pc
AU$Pd = $Pd
AU$Pe = $Pe
AU$Pf = $Pf
AU$Pg = $Pg
AU$Ph = $Ph
AU$Pi = $Pi
AU$Pj = $Pj

IF(AU==ABSENT() or AU<0)  AU=0  end
IF(AU$Pa==ABSENT() or AU$Pa<0)  AU$Pa=0  end
IF(AU$Pb==ABSENT() or AU$Pb<0)  AU$Pb=0  end
IF(AU$Pc==ABSENT() or AU$Pc<0)  AU$Pc=0  end
IF(AU$Pd==ABSENT() or AU$Pd<0)  AU$Pd=0  end
IF(AU$Pe==ABSENT() or AU$Pe<0)  AU$Pe=0  end
IF(AU$Pf==ABSENT() or AU$Pf<0)  AU$Pf=0  end
IF(AU$Pg==ABSENT() or AU$Pg<0)  AU$Pg=0  end
IF(AU$Ph==ABSENT() or AU$Ph<0)  AU$Ph=0  end
IF(AU$Pi==ABSENT() or AU$Pi<0)  AU$Pi=0  end
IF(AU$Pj==ABSENT() or AU$Pj<0)  AU$Pj=0  end

erase($Pa,$Pb,$Pc,$Pd,$Pe)
erase($Pf,$Pg,$Ph,$Pi,$Pj)

GO

!EXTRA    &IN(T5),&OUT(T6)

VOLUME=XINC*YINC*ZINC
TONNES=VOLUME*DENSITY

AG=0  AGT=0     AGM=0  ORE=0 MET=0
ORE$Pa=0    MET$Pa=0   ORE$Pb=0    MET$Pb=0
ORE$Pc=0    MET$Pc=0   ORE$Pd=0    MET$Pd=0
ORE$Pe=0    MET$Pe=0   ORE$Pf=0    MET$Pf=0
ORE$Pg=0    MET$Pg=0   ORE$Ph=0    MET$Ph=0
ORE$Pi=0    MET$Pi=0   ORE$Pj=0   MET$Pj=0

TON=VOLUME*DENSITY

if(AU>=$PITOFF) ORE=TONNES MET=ORE*AU END

if(AU$Pa>=$PITOFF) ORE$Pa=TONNES MET$Pa=ORE$Pa*AU$Pa END
if(AU$Pb>=$PITOFF) ORE$Pb=TONNES MET$Pb=ORE$Pb*AU$Pb END
if(AU$Pc>=$PITOFF) ORE$Pc=TONNES MET$Pc=ORE$Pc*AU$Pc END
if(AU$Pd>=$PITOFF) ORE$Pd=TONNES MET$Pd=ORE$Pd*AU$Pd END
if(AU$Pe>=$PITOFF) ORE$Pe=TONNES MET$Pe=ORE$Pe*AU$Pe END
if(AU$Pf>=$PITOFF) ORE$Pf=TONNES MET$Pf=ORE$Pf*AU$Pf END
if(AU$Pg>=$PITOFF) ORE$Pg=TONNES MET$Pg=ORE$Pg*AU$Pg END
if(AU$Ph>=$PITOFF) ORE$Ph=TONNES MET$Ph=ORE$Ph*AU$Ph END
if(AU$Pi>=$PITOFF) ORE$Pi=TONNES MET$Pi=ORE$Pi*AU$Pi END
if(AU$Pj>=$PITOFF) ORE$Pj=TONNES MET$Pj=ORE$Pj*AU$Pj END

IX=INT(IJK/(NY*NZ))
N=IJK-IX*NY*NZ
IY=INT(N/NZ)
IZ=N-IY*NZ
BENCH=ZMORIG+(IZ*$SZINC#)-$LEVEL
XC=XMORIG+(IX*$SXINC#)+(0.5*$SXINC#)-$EAST
YC=YMORIG+(IY*$SYINC#)+(0.5*$SYINC#)-$NORTH

erase(IX,N,IY,IZ)
erase(AU$Pa,AU$Pb,AU$Pc,AU$Pd,AU$Pe)
erase(AU$Pf,AU$Pg,AU$Ph,AU$Pi,AU$Pj)
```

```
erase(IJK,XINC,YINC,ZINC,AU,DENSITY,VOLUME)
erase(XMORIG,YMORIG,ZMORIG,NX,NY,NZ,AG,ZC)

GO

!MGSORT   &IN(T6),&OUT(T7),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),*KEY4(YC),
        @ORDER=1.0

!ACCMLT   &IN(T7),&OUT($RUN),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),*KEY4(YC),
        @ALLRECS=0.0,@UNSORTED=0.0

!GOTO $RETURN#

!FINISH:REM finish

!JOIN&IN1(RUN1),&IN2(RUN2),&OUT(T8),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),
*KEY4(YC),@SUBSETR=0.0,@SUBSETF=0.0,@CARTJOIN=0.0

!JOIN&IN1(RUN3),&IN2(RUN4),&OUT(T8a),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),*KEY4(YC),@SUBS
ETR=0.0,@SUBSETF=0.0,@CARTJOIN=0.0

!JOIN&IN1(RUN5),&IN2(RUN6),&OUT(T8b),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),*KEY4(YC),@SUBS
ETR=0.0,@SUBSETF=0.0,@CARTJOIN=0.0

!JOIN&IN1(RUN7),&IN2(RUN8),&OUT(T8c),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),*KEY4(YC),@SUBS
ETR=0.0,@SUBSETF=0.0,@CARTJOIN=0.0

!JOIN&IN1(RUN9),&IN2(RUN10),&OUT(T8d),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),*KEY4(YC),@SUB
SETR=0.0,@SUBSETF=0.0,@CARTJOIN=0.0

!JOIN&IN1(T8c),&IN2(T8d),&OUT(T8da),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),*KEY4(YC),@SUBSET
R=0.0,@SUBSETF=0.0,@CARTJOIN=0.0

!JOIN &IN1(T8),&IN2(T8a),&OUT(T9a),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),
        *KEY4(YC),@SUBSETR=0.0,@SUBSETF=0.0,@CARTJOIN=0.0


!JOIN&IN1(T9a),&IN2(T8b),&OUT(T9c),*KEY1(PIT),*KEY2(BENCH),*KEY3(XC),*KEY4(YC),@SUBSETR
=0.0,@SUBSETF=0.0,@CARTJOIN=0.0

!EXTRA    &IN(T9c),&OUT(T11)
OP=$Op#
GO

!EXTRA    &IN(T8da),&OUT(T10)
OP=$Op#
GO

!MGSORT&IN(T11),&OUT(T12),*KEY1(OP),*KEY2(PIT),*KEY3(BENCH),*KEY4(XC),*KEY5(YC),@ORDE
R=2.0

!MGSORT&IN(T10),&OUT(T13),*KEY1(OP),*KEY2(PIT),*KEY3(BENCH),*KEY4(XC),*KEY5(YC),@ORDE
R=2.0

!OUTPUT   &IN(T12),@CSV=1.0,@NODD=0.0
$folder\XPAC_$Option_1-60.csv

!OUTPUT   &IN(T13),@CSV=1.0,@NODD=0.0
$folder\XPAC_$Option_60-100.csv

!OPSYS
del T1.dm T2.dm T3.dm T4.dm T6.dm T5.dm
del RUN1.dm RUN2.dm RUN3.dm RUN4.dm
del T7.dm T8.dm T9.dm T10.dm T11.dm
del T8a.dm T8b.dm T8c.dm T8d.dm
del T9a.dm T9b.dm T9c.dm T13.dm
del TPROT.dm _field.dm T12.dm
```

```
del RUN5.dm RUN6.dm RUN7.dm RUN8.dm
del RUN9.dm RUN10.dm
!END
```

Appendix 8: Cumulative distribution data for Geita, Sadiola and Morila Mines

Cumulative distribution processed ounces data for case study mines for Options 1 to 3

GOLD	MORILA			SADIOLA			GEITA			CVSA		
PROBABILITY	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3
1	993	1436	413	7130	2469.45	1661.97	2927	2469	2305	993	1436	413
2	994	1439	414	7139	2472.14	1664.26	2950	2472	2333	994	1439	414
3	995	1440	414	7149	2473.1	1664.94	3014	2473	2337	995	1440	414
4	1001	1442	415	7150	2474.15	1664.97	3062	2474	2338	1001	1442	415
5	1001	1442	415	7150	2474.47	1664.98	3070	2474	2343	1001	1442	415
6	1001	1446	415	7154	2474.56	1666.19	3092	2475	2348	1001	1446	415
7	1001	1446	416	7155	2474.84	1666.94	3102	2475	2356	1001	1446	416
8	1002	1448	417	7156	2474.96	1667.19	3107	2475	2363	1002	1448	417
9	1003	1449	418	7159	2475.1	1667.35	3127	2475	2363	1003	1449	418
10	1005	1457	418	7159	2475.38	1667.41	3252	2475	2364	1005	1457	418
11	1007	1461	419	7160	2475.66	1667.88	3866	2476	2367	1007	1461	419
12	1008	1467	419	7162	2475.69	1667.94	3884	2476	2370	1008	1467	419
13	1014	1468	420	7162	2475.94	1667.96	3911	2476	2373	1014	1468	420
14	1014	1470	423	7164	2476.09	1667.98	3940	2476	2373	1014	1470	423
15	1014	1471	424	7164	2476.3	1668.33	3955	2476	2380	1014	1471	424
16	1018	1471	425	7165	2476.54	1668.57	3966	2477	2382	1018	1471	425
17	1019	1472	426	7169	2476.62	1668.58	3987	2477	2386	1019	1472	426
18	1020	1472	426	7170	2476.77	1668.62	3991	2477	2390	1020	1472	426
19	1020	1478	426	7170	2476.83	1668.76	4001	2477	2398	1020	1478	426
20	1021	1478	426	7171	2476.89	1668.9	4004	2477	2403	1021	1478	426
21	1021	1483	427	7171	2476.95	1669.14	4012	2477	2403	1021	1483	427
22	1022	1484	427	7172	2477.51	1669.45	4024	2478	2405	1022	1484	427
23	1023	1487	427	7173	2477.6	1669.89	4025	2478	2406	1023	1487	427
24	1026	1487	428	7174	2477.77	1669.94	4038	2478	2407	1026	1487	428
25	1027	1487	428	7175	2477.81	1670.33	4040	2478	2407	1027	1487	428
26	1027	1488	428	7176	2477.96	1670.42	4042	2478	2412	1027	1488	428
27	1027	1488	429	7176	2478.08	1670.44	4043	2478	2412	1027	1488	429
28	1028	1493	429	7180	2478.36	1670.56	4046	2478	2413	1028	1493	429
29	1031	1494	430	7181	2478.47	1670.56	4046	2478	2418	1031	1494	430
30	1031	1495	430	7181	2478.87	1670.57	4056	2479	2419	1031	1495	430
31	1032	1495	430	7182	2478.89	1670.65	4063	2479	2421	1032	1495	430
32	1032	1496	431	7183	2478.96	1670.8	4063	2479	2430	1032	1496	431
33	1032	1496	431	7183	2479.37	1670.89	4066	2479	2430	1032	1496	431
34	1033	1496	432	7185	2479.42	1670.91	4068	2479	2430	1033	1496	432
35	1033	1496	432	7186	2479.84	1670.94	4075	2480	2435	1033	1496	432
36	1033	1498	434	7187	2480.18	1670.94	4079	2480	2436	1033	1498	434
37	1035	1498	434	7188	2480.29	1670.95	4081	2480	2436	1035	1498	434
38	1035	1500	434	7189	2481.27	1671.03	4081	2481	2438	1035	1500	434
39	1035	1501	434	7189	2481.58	1671.18	4090	2482	2438	1035	1501	434
40	1036	1503	434	7189	2486.02	1671.78	4091	2486	2440	1036	1503	434
41	1036	1505	434	7192	2498.11	1671.84	4098	2498	2445	1036	1505	434
42	1037	1505	435	7192	2498.34	1671.89	4103	2498	2446	1037	1505	435
43	1039	1505	435	7193	2499.07	1672.11	4105	2499	2447	1039	1505	435
44	1039	1506	435	7193	2499.25	1672.25	4107	2499	2459	1039	1506	435
45	1039	1507	436	7193	2499.87	1672.46	4109	2500	2461	1039	1507	436
46	1040	1507	436	7194	2500	1672.63	4112	2500	2471	1040	1507	436
47	1040	1508	436	7194	2500.03	1672.72	4114	2500	2472	1040	1508	436
48	1040	1508	437	7195	2500.42	1672.75	4122	2500	2473	1040	1508	437
49	1040	1509	437	7196	2500.45	1672.85	4130	2500	2473	1040	1509	437
50	1041	1511	437	7197	2500.48	1672.89	4131	2500	2475	1041	1511	437
51	1042	1511	437	7198	2500.58	1672.91	4138	2501	2475	1042	1511	437
52	1043	1511	438	7199	2500.65	1672.98	4138	2501	2476	1043	1511	438
53	1044	1511	438	7200	2500.67	1673.09	4144	2501	2480	1044	1511	438
54	1044	1512	439	7201	2501	1673.27	4150	2501	2480	1044	1512	439
55	1046	1512	439	7201	2501.13	1673.34	4151	2501	2483	1046	1512	439
56	1046	1513	440	7201	2501.27	1673.34	4163	2501	2485	1046	1513	440
57	1046	1514	440	7203	2501.31	1673.36	4163	2501	2489	1046	1514	440
58	1047	1515	440	7203	2501.33	1673.44	4163	2501	2490	1047	1515	440
59	1047	1518	440	7204	2501.37	1673.55	4166	2501	2494	1047	1518	440
60	1050	1520	441	7204	2501.38	1673.73	4168	2501	2501	1050	1520	441
61	1050	1523	441	7204	2501.44	1673.85	4172	2501	2502	1050	1523	441
62	1051	1525	441	7207	2501.52	1673.95	4172	2502	2503	1051	1525	441
63	1052	1526	441	7209	2501.69	1674.28	4175	2502	2503	1052	1526	441
64	1052	1526	441	7209	2501.69	1674.43	4178	2502	2507	1052	1526	441
65	1053	1528	441	7209	2501.83	1674.49	4182	2502	2508	1053	1528	441
66	1053	1528	442	7210	2501.86	1674.51	4184	2502	2509	1053	1528	442
67	1054	1528	443	7211	2501.9	1674.59	4193	2502	2509	1054	1528	443
68	1054	1528	443	7211	2501.98	1674.65	4196	2502	2516	1054	1528	443
69	1054	1528	443	7214	2502	1674.77	4207	2502	2517	1054	1528	443
70	1056	1529	443	7215	2502.05	1674.91	4214	2502	2519	1056	1529	443
71	1057	1529	444	7217	2502.15	1674.99	4214	2502	2524	1057	1529	444
72	1057	1531	444	7218	2502.28	1675.04	4217	2502	2527	1057	1531	444
73	1058	1532	444	7219	2502.56	1675.34	4220	2503	2529	1058	1532	444
74	1058	1533	445	7220	2502.57	1675.6	4221	2503	2529	1058	1533	445

GOLD	MORILA			SADIOLA			GEITA			CVSA		
PROBABILITY	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3
75	1058	1533	446	7222	2502.62	1675.76	4222	2503	2536	1058	1533	446
76	1061	1539	446	7222	2502.66	1675.82	4229	2503	2537	1061	1539	446
77	1063	1541	447	7223	2502.66	1675.84	4230	2503	2539	1063	1541	447
78	1063	1542	447	7224	2502.68	1675.84	4230	2503	2540	1063	1542	447
79	1064	1542	447	7225	2503.15	1675.98	4232	2503	2543	1064	1542	447
80	1065	1542	447	7225	2503.17	1676.15	4232	2503	2546	1065	1542	447
81	1066	1542	449	7226	2503.18	1676.31	4233	2503	2548	1066	1542	449
82	1067	1547	449	7226	2503.24	1676.34	4237	2503	2557	1067	1547	449
83	1069	1547	449	7227	2503.43	1676.39	4243	2503	2558	1069	1547	449
84	1069	1551	449	7228	2503.44	1676.96	4255	2503	2566	1069	1551	449
85	1069	1552	450	7232	2503.45	1677.33	4256	2503	2577	1069	1552	450
86	1071	1553	450	7232	2503.52	1678.15	4263	2504	2586	1071	1553	450
87	1075	1553	451	7234	2503.69	1678.57	4272	2504	2595	1075	1553	451
88	1076	1553	451	7236	2503.7	1678.89	4284	2504	2599	1076	1553	451
89	1078	1555	452	7236	2503.8	1678.98	4287	2504	2604	1078	1555	452
90	1078	1558	454	7238	2504.12	1679.27	4290	2504	2607	1078	1558	454
91	1082	1558	454	7238	2504.13	1680.16	4307	2504	2609	1082	1558	454
92	1084	1564	455	7238	2504.29	1680.47	4310	2504	2609	1084	1564	455
93	1084	1565	455	7246	2504.58	1680.64	4313	2505	2612	1084	1565	455
94	1085	1570	456	7248	2504.89	1681.09	4321	2505	2623	1085	1570	456
95	1085	1576	457	7250	2505.26	1681.12	4321	2505	2630	1085	1576	457
96	1086	1580	457	7253	2505.7	1681.54	4332	2506	2633	1086	1580	457
97	1086	1583	458	7260	2506.14	1682.1	4344	2506	2634	1086	1583	458
98	1090	1585	462	7264	2506.25	1682.2	4354	2506	2640	1090	1585	462
99	1092	1586	463	7271	2506.33	1683.23	4388	2506	2645	1092	1586	463
99.5	1101	1608	469		2508.5	1685.79	4397	2509	2653	1101	1608	469

Cumulative distribution Grade data for case study mines for Options 1 to 3

GRADE	MORILA			SADIOLA			GEITA			CVSA		
PROBABILITY	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3
1	6.79	5.84	5.66	2.13	3.25	2.25	1.88	1.94	2.42	8.68	9.32	7.63
2	6.84	5.85	5.67	2.13	3.25	2.25	1.89	1.96	2.45	8.68	9.33	7.64
3	6.87	5.86	5.68	2.13	3.25	2.25	1.91	1.97	2.45	8.70	9.34	7.65
4	6.90	5.87	5.70	2.13	3.25	2.25	1.91	1.98	2.46	8.73	9.38	7.65
5	6.91	5.87	5.71	2.13	3.26	2.25	1.91	1.98	2.46	8.75	9.38	7.66
6	6.91	5.88	5.74	2.13	3.26	2.26	1.91	1.98	2.47	8.77	9.41	7.66
7	6.92	5.94	5.75	2.13	3.26	2.26	1.92	1.98	2.47	8.79	9.41	7.68
8	6.94	5.94	5.75	2.13	3.26	2.26	1.92	1.99	2.48	8.80	9.41	7.69
9	6.96	5.95	5.77	2.13	3.26	2.26	1.92	1.99	2.48	8.81	9.43	7.71
10	6.96	5.96	5.78	2.13	3.26	2.26	1.92	1.99	2.48	8.82	9.45	7.72
11	7.00	5.96	5.79	2.13	3.26	2.26	1.92	1.99	2.49	8.82	9.47	7.72
12	7.00	5.97	5.80	2.13	3.26	2.26	1.92	2.00	2.49	8.83	9.48	7.73
13	7.00	5.98	5.80	2.13	3.26	2.26	1.92	2.00	2.49	8.83	9.48	7.74
14	7.01	5.99	5.80	2.13	3.26	2.26	1.92	2.00	2.49	8.83	9.49	7.81
15	7.06	5.99	5.81	2.13	3.26	2.26	1.93	2.00	2.50	8.85	9.49	7.83
16	7.09	6.00	5.84	2.13	3.26	2.26	1.93	2.00	2.50	8.86	9.50	7.83
17	7.11	6.02	5.87	2.13	3.26	2.26	1.93	2.00	2.51	8.90	9.51	7.85
18	7.12	6.02	5.87	2.13	3.26	2.26	1.93	2.00	2.51	8.90	9.51	7.86
19	7.12	6.05	5.88	2.13	3.26	2.26	1.94	2.00	2.52	8.90	9.52	7.87
20	7.14	6.05	5.88	2.13	3.26	2.26	1.94	2.00	2.52	8.90	9.54	7.87
21	7.14	6.06	5.88	2.13	3.26	2.26	1.94	2.00	2.52	8.91	9.57	7.87
22	7.15	6.06	5.89	2.13	3.26	2.26	1.94	2.01	2.53	8.91	9.58	7.87
23	7.15	6.06	5.89	2.13	3.26	2.26	1.94	2.01	2.53	8.91	9.58	7.88
24	7.15	6.07	5.89	2.13	3.26	2.26	1.95	2.01	2.53	8.91	9.59	7.89
25	7.16	6.07	5.90	2.13	3.26	2.26	1.95	2.01	2.53	8.92	9.61	7.89
26	7.17	6.08	5.91	2.13	3.26	2.26	1.95	2.01	2.53	8.92	9.62	7.90
27	7.17	6.10	5.91	2.13	3.26	2.26	1.95	2.01	2.53	8.93	9.64	7.91
28	7.17	6.10	5.93	2.14	3.26	2.26	1.95	2.01	2.53	8.94	9.65	7.92
29	7.17	6.10	5.93	2.14	3.26	2.26	1.95	2.01	2.54	8.95	9.65	7.93
30	7.18	6.12	5.94	2.14	3.26	2.26	1.95	2.01	2.54	8.95	9.66	7.93
31	7.18	6.12	5.94	2.14	3.26	2.26	1.95	2.01	2.54	8.95	9.66	7.93
32	7.18	6.13	5.94	2.14	3.26	2.26	1.95	2.02	2.55	8.95	9.67	7.94
33	7.18	6.13	5.95	2.14	3.26	2.26	1.96	2.02	2.55	8.95	9.67	7.95
34	7.19	6.13	5.95	2.14	3.26	2.26	1.96	2.02	2.55	8.97	9.68	7.96
35	7.20	6.13	5.96	2.14	3.26	2.26	1.96	2.02	2.56	8.97	9.69	7.97
36	7.21	6.14	5.96	2.14	3.26	2.26	1.96	2.02	2.56	8.98	9.69	8.00
37	7.21	6.15	5.97	2.14	3.26	2.26	1.96	2.02	2.56	8.98	9.70	8.00
38	7.21	6.15	5.97	2.14	3.26	2.26	1.96	2.02	2.56	8.98	9.71	8.00
39	7.22	6.15	5.98	2.14	3.26	2.26	1.96	2.02	2.56	8.98	9.72	8.00
40	7.24	6.15	5.98	2.14	3.26	2.26	1.96	2.02	2.56	8.99	9.72	8.01

GRADE	MORILA			SADIOLA			GEITA			CVSA		
PROBABILITY	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3
41	7.26	6.16	5.98	2.14	3.26	2.26	1.97	2.02	2.57	9.01	9.72	8.01
42	7.26	6.16	5.98	2.14	3.26	2.26	1.97	2.02	2.57	9.01	9.72	8.02
43	7.26	6.16	5.99	2.14	3.26	2.26	1.97	2.02	2.57	9.03	9.73	8.03
44	7.26	6.17	6.00	2.14	3.26	2.26	1.97	2.02	2.58	9.03	9.73	8.03
45	7.27	6.17	6.00	2.14	3.26	2.26	1.98	2.02	2.58	9.05	9.74	8.04
46	7.27	6.17	6.00	2.14	3.26	2.26	1.98	2.02	2.60	9.05	9.74	8.04
47	7.27	6.17	6.00	2.14	3.26	2.26	1.98	2.02	2.60	9.06	9.75	8.04
48	7.28	6.18	6.00	2.14	3.26	2.26	1.98	2.03	2.60	9.08	9.75	8.05
49	7.29	6.18	6.01	2.14	3.26	2.26	1.98	2.03	2.60	9.09	9.76	8.05
50	7.30	6.19	6.02	2.14	3.26	2.26	1.98	2.03	2.60	9.09	9.76	8.07
51	7.30	6.20	6.02	2.14	3.26	2.26	1.98	2.03	2.60	9.10	9.76	8.07
52	7.30	6.20	6.02	2.14	3.26	2.26	1.98	2.03	2.60	9.10	9.76	8.07
53	7.30	6.21	6.02	2.14	3.26	2.26	1.98	2.03	2.60	9.11	9.77	8.08
54	7.32	6.21	6.03	2.14	3.26	2.26	1.98	2.03	2.60	9.11	9.77	8.09
55	7.32	6.21	6.03	2.14	3.26	2.26	1.99	2.03	2.61	9.11	9.78	8.10
56	7.32	6.21	6.03	2.14	3.26	2.26	1.99	2.03	2.61	9.11	9.79	8.11
57	7.32	6.23	6.04	2.14	3.26	2.27	1.99	2.04	2.61	9.12	9.79	8.11
58	7.32	6.24	6.04	2.14	3.26	2.27	1.99	2.04	2.61	9.12	9.79	8.12
59	7.33	6.24	6.05	2.14	3.26	2.27	1.99	2.04	2.62	9.13	9.81	8.12
60	7.34	6.24	6.06	2.14	3.26	2.27	1.99	2.04	2.63	9.14	9.81	8.13
61	7.35	6.24	6.07	2.14	3.3	2.27	1.99	2.04	2.63	9.15	9.82	8.13
62	7.35	6.26	6.07	2.14	3.3	2.27	1.99	2.04	2.63	9.15	9.83	8.13
63	7.36	6.26	6.08	2.14	3.3	2.27	1.99	2.04	2.63	9.15	9.84	8.14
64	7.37	6.26	6.08	2.14	3.3	2.27	2.00	2.04	2.63	9.16	9.84	8.14
65	7.37	6.26	6.08	2.14	3.3	2.27	2.00	2.04	2.63	9.16	9.86	8.14
66	7.38	6.27	6.09	2.14	3.3	2.27	2.00	2.04	2.63	9.16	9.87	8.16
67	7.39	6.27	6.09	2.14	3.3	2.27	2.00	2.04	2.63	9.17	9.88	8.16
68	7.39	6.27	6.10	2.14	3.3	2.27	2.00	2.04	2.64	9.17	9.88	8.17
69	7.41	6.27	6.10	2.14	3.3	2.27	2.01	2.04	2.64	9.17	9.89	8.17
70	7.41	6.28	6.10	2.14	3.3	2.27	2.01	2.04	2.65	9.17	9.91	8.17
71	7.42	6.28	6.10	2.14	3.3	2.27	2.01	2.04	2.65	9.17	9.91	8.18
72	7.43	6.28	6.12	2.14	3.3	2.27	2.01	2.05	2.65	9.20	9.92	8.19
73	7.43	6.29	6.13	2.14	3.3	2.27	2.01	2.05	2.66	9.20	9.92	8.19
74	7.44	6.30	6.13	2.14	3.3	2.27	2.01	2.05	2.66	9.21	9.93	8.21
75	7.45	6.31	6.13	2.14	3.3	2.27	2.01	2.05	2.66	9.22	9.93	8.23
76	7.45	6.31	6.14	2.14	3.3	2.27	2.01	2.05	2.66	9.23	9.94	8.23
77	7.48	6.32	6.15	2.14	3.3	2.27	2.01	2.05	2.67	9.23	9.94	8.24
78	7.48	6.32	6.16	2.14	3.3	2.27	2.02	2.05	2.67	9.23	9.94	8.25
79	7.49	6.35	6.16	2.14	3.3	2.27	2.02	2.05	2.67	9.23	9.95	8.25
80	7.49	6.36	6.17	2.14	3.3	2.27	2.02	2.05	2.67	9.23	9.95	8.25
81	7.50	6.36	6.17	2.14	3.3	2.27	2.03	2.05	2.68	9.24	9.96	8.28
82	7.50	6.37	6.18	2.14	3.3	2.27	2.03	2.05	2.69	9.25	9.97	8.28
83	7.51	6.37	6.19	2.14	3.3	2.27	2.04	2.06	2.69	9.26	9.98	8.28
84	7.52	6.37	6.19	2.14	3.3	2.27	2.04	2.06	2.69	9.26	10.01	8.29
85	7.52	6.38	6.20	2.14	3.3	2.27	2.04	2.06	2.71	9.26	10.01	8.31
86	7.52	6.39	6.20	2.14	3.3	2.27	2.04	2.06	2.72	9.28	10.02	8.31
87	7.54	6.40	6.22	2.14	3.3	2.27	2.05	2.07	2.72	9.29	10.03	8.31
88	7.54	6.41	6.22	2.14	3.3	2.27	2.06	2.07	2.73	9.29	10.03	8.32
89	7.55	6.41	6.22	2.14	3.31	2.27	2.06	2.08	2.73	9.32	10.06	8.33
90	7.57	6.41	6.23	2.14	3.31	2.27	2.07	2.09	2.74	9.33	10.07	8.37
91	7.58	6.41	6.24	2.14	3.31	2.27	2.71	2.82	2.74	9.34	10.07	8.37
92	7.58	6.42	6.26	2.14	3.31	2.27	2.78	2.84	2.74	9.34	10.07	8.40
93	7.60	6.45	6.26	2.14	3.31	2.27	2.82	2.84	2.74	9.34	10.08	8.40
94	7.63	6.45	6.27	2.15	3.31	2.28	2.87	2.86	2.75	9.36	10.11	8.42
95	7.63	6.46	6.28	2.15	3.31	2.28	2.89	2.90	2.76	9.40	10.13	8.42
96	7.64	6.46	6.29	2.15	3.31	2.28	2.92	2.90	2.76	9.42	10.15	8.43
97	7.65	6.47	6.30	2.15	3.31	2.28	2.92	2.93	2.77	9.43	10.16	8.45
98	7.72	6.54	6.38	2.15	3.31	2.28	2.95	2.94	2.77	9.48	10.19	8.53
99	7.77	6.55	6.39	2.15	3.31	2.28	3.00	2.95	2.78	9.50	10.37	8.54
99.5	7.80	6.66	6.51	2.15	3.31	2.28	3.06	2.97	2.79	9.62	10.43	8.65

Cumulative distribution NPV data for case study mines for Options 1 to 3

NPV	MORILA			SADIOLA			GEITA			CVSA		
PROBABILITY	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3
1	4257	3182	3959	2199	892	273	981	142	91	773	692	302
2	4324	3224	3980	2203	1132.92	276	1007	991	93	774	699	303
3	4337	3278	3997	2206	1132.98	276	1024	1048	97	777	701	303
4	4346	3333	4040	2208	1133.06	276	1026	1050	100	780	703	304
5	4375	3334	4050	2208	1133.41	277	1038	1051	101	781	703	304
6	4410	3343	4050	2208	1133.78	278	1040	1086	101	781	705	305
7	4418	3362	4056	2209	1133.82	278	1055	1091	107	781	705	306
8	4418	3379	4059	2209	1134.28	278	1067	1094	108	783	707	306
9	4422	3397	4060	2209	1134.41	279	1068	1098	109	785	710	307
10	4433	3416	4060	2209	1134.43	279	1081	1123	109	786	711	307
11	4435	3417	4072	2210	1134.77	279	1089	1143	109	787	712	308
12	4437	3418	4082	2210	1134.85	280	1106	1195	110	790	712	308
13	4454	3433	4095	2210	1134.87	280	1135	1234	111	793	713	309
14	4485	3443	4096	2211	1134.92	280	1137	1245	114	794	713	314
15	4489	3447	4132	2211	1134.96	280	1140	1254	114	796	715	314
16	4509	3448	4149	2212	1135.11	280	1143	1254	115	796	716	314
17	4547	3451	4165	2212	1135.2	280	1145	1258	119	797	716	315
18	4548	3455	4183	2213	1135.28	280	1146	1259	124	798	717	315
19	4559	3462	4188	2213	1135.32	280	1148	1260	125	799	722	315
20	4564	3467	4211	2213	1135.33	280	1150	1267	125	799	723	316
21	4564	3479	4211	2214	1135.55	281	1151	1268	126	799	724	316
22	4571	3485	4213	2215	1135.57	281	1151	1268	127	800	724	317
23	4575	3491	4215	2216	1135.62	282	1155	1269	127	803	726	317
24	4581	3492	4224	2216	1135.67	282	1157	1271	129	804	726	318
25	4585	3493	4225	2216	1135.69	282	1158	1273	130	804	727	318
26	4588	3499	4229	2216	1135.75	282	1159	1275	132	806	727	318
27	4589	3513	4231	2217	1135.82	282	1160	1275	132	807	727	318
28	4596	3530	4232	2217	1135.84	282	1164	1276	133	807	728	319
29	4605	3531	4243	2217	1135.88	282	1166	1279	133	810	728	319
30	4606	3531	4248	2218	1136.02	282	1169	1280	133	810	729	320
31	4615	3539	4251	2218	1136.11	283	1172	1280	135	811	729	320
32	4619	3540	4251	2219	1136.15	283	1177	1286	135	812	729	321
33	4630	3541	4261	2219	1136.19	283	1178	1295	137	812	730	321
34	4637	3547	4265	2219	1136.23	283	1181	1296	140	813	731	321
35	4645	3555	4273	2219	1136.26	283	1186	1297	141	813	731	321
36	4646	3561	4276	2219	1136.32	283	1187	1298	142	815	732	323
37	4648	3573	4277	2220	1136.5	283	1187	1299	142	815	732	324
38	4648	3576	4280	2220	1136.52	283	1191	1299	142	816	732	324
39	4652	3580	4282	2221	1136.53	283	1195	1300	145	816	732	324
40	4656	3580	4284	2221	1136.7	283	1195	1300	146	817	733	324
41	4660	3580	4285	2221	1136.82	283	1198	1301	147	817	734	325
42	4666	3584	4293	2221	1136.83	283	1199	1306	147	817	735	325
43	4670	3586	4307	2221	1136.87	284	1200	1310	149	817	736	325
44	4672	3587	4309	2222	1136.95	284	1201	1310	149	818	736	326
45	4683	3594	4313	2222	1137.08	284	1201	1311	150	818	737	326
46	4683	3595	4320	2223	1137.11	285	1202	1312	150	818	738	326
47	4686	3596	4324	2224	1137.12	285	1203	1316	151	821	738	326
48	4686	3597	4325	2224	1137.12	285	1203	1317	152	821	738	327
49	4691	3597	4326	2224	1137.13	285	1206	1318	153	822	740	327
50	4701	3600	4326	2224	1137.19	285	1206	1318	153	823	741	327
51	4714	3602	4327	2225	1137.41	285	1207	1318	155	824	741	327
52	4723	3605	4330	2225	1137.51	285	1207	1321	155	824	742	329
53	4733	3608	4330	2225	1137.52	285	1208	1321	157	825	742	330
54	4739	3611	4331	2226	1137.6	285	1210	1324	157	826	742	330
55	4746	3612	4339	2226	1137.61	285	1212	1324	157	827	743	330
56	4761	3615	4341	2226	1137.65	285	1212	1324	158	827	744	330
57	4762	3615	4348	2226	1137.8	285	1216	1324	165	827	745	330
58	4766	3619	4351	2226	1137.8	286	1216	1326	166	827	746	330
59	4772	3630	4356	2226	1137.86	286	1217	1327	167	828	747	330
60	4780	3634	4357	2226	1137.88	286	1218	1328	170	828	747	330
61	4782	3637	4358	2227	1138.01	286	1218	1332	170	829	747	331
62	4783	3638	4362	2227	1138.07	286	1221	1334	170	829	748	331
63	4785	3641	4379	2227	1138.51	286	1225	1334	171	830	748	331
64	4792	3648	4391	2227	1138.51	286	1226	1334	171	830	748	332
65	4796	3651	4392	2227	1138.53	286	1229	1338	172	830	748	332
66	4802	3652	4398	2227	1138.54	286	1230	1338	173	832	749	333
67	4806	3654	4398	2227	1138.7	286	1230	1339	174	833	749	333
68	4810	3655	4405	2228	1138.73	287	1231	1340	174	834	751	333
69	4812	3657	4408	2229	1138.92	287	1233	1341	181	837	752	334
70	4816	3658	4429	2229	1139.23	287	1233	1342	182	837	754	334
71	4838	3664	4430	2230	1139.45	287	1240	1344	183	837	754	334
72	4856	3667	4444	2230	1139.59	287	1242	1348	183	837	754	334
73	4856	3677	4461	2230	1139.63	288	1243	1349	184	837	755	335
74	4876	3677	4461	2230	1139.64	288	1243	1349	187	838	755	335

NPV	MORILA			SADIOLA			GEITA			CVSA		
PROBABILITY	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3
75	4882	3679	4464	2232	1139.73	288	1245	1352	187	838	756	336
76	4889	3681	4464	2232	1139.75	288	1247	1352	188	839	756	337
77	4890	3689	4466	2234	1139.77	288	1249	1356	189	841	757	337
78	4894	3695	4473	2234	1139.82	288	1251	1360	191	841	757	338
79	4896	3704	4480	2234	1139.86	288	1254	1362	191	841	757	338
80	4897	3716	4482	2234	1140.23	288	1254	1362	192	842	761	338
81	4898	3717	4493	2234	1140.38	288	1255	1362	192	845	762	339
82	4902	3718	4499	2235	1140.57	288	1257	1365	196	845	763	340
83	4906	3718	4522	2236	1140.61	289	1261	1366	199	845	764	340
84	4907	3721	4529	2236	1140.65	289	1265	1366	205	848	764	340
85	4919	3728	4546	2236	1140.87	289	1266	1367	207	849	764	340
86	4948	3730	4550	2239	1141.12	290	1266	1367	211	850	765	341
87	4960	3731	4551	2239	1141.25	291	1270	1371	211	851	766	342
88	4970	3747	4552	2239	1141.27	291	1271	1375	216	855	766	342
89	4977	3750	4564	2240	1141.38	291	1271	1375	218	857	766	343
90	4984	3759	4565	2242	1141.72	291	1272	1375	218	857	767	343
91	4996	3760	4572	2243	1142.03	293	1272	1377	218	857	767	345
92	5007	3784	4591	2243	1142.03	293	1273	1380	223	861	770	346
93	5007	3786	4596	2245	1142.6	293	1276	1388	226	862	774	346
94	5018	3801	4602	2245	1142.68	293	1286	1389	231	864	778	346
95	5040	3823	4629	2246	1143.38	293	1288	1394	232	865	779	347
96	5057	3825	4635	2249	1143.62	294	1304	1399	232	865	780	348
97	5115	3853	4689	2252	1143.71	295	1316	1405	233	867	780	348
98	5115	3899	4712	2254	1143.89	295	1318	1411	236	867	781	353
99	5120	3921	4750	2260	1144.08	296	1321	1422	237	868	791	353
99.5	5146	4010	4854	2268	1148.36	298	1322	1442	242	882	794	360

Cumulative distribution Gold price to cost for case study mines for Options 1 to 3

Gold price per cost	MORILA			SADIOLA			GEITA			CVSA		
PROBABILITY	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3
1	3.96	3.83	3.69	1.93	2.18	1.2	1.6	1.53	1.13	4.20	5.04	3.26
2	4.00	3.84	3.70	1.93	2.18	1.21	1.61	1.54	1.14	4.20	5.04	3.26
3	4.01	3.85	3.71	1.93	2.18	1.21	1.61	1.54	1.15	4.21	5.06	3.26
4	4.03	3.86	3.72	1.93	2.18	1.21	1.61	1.54	1.15	4.22	5.06	3.27
5	4.04	3.87	3.73	1.93	2.18	1.21	1.62	1.54	1.15	4.23	5.07	3.27
6	4.05	3.90	3.75	1.93	2.18	1.21	1.62	1.55	1.15	4.23	5.07	3.27
7	4.07	3.90	3.76	1.93	2.18	1.21	1.63	1.55	1.16	4.23	5.08	3.28
8	4.07	3.91	3.76	1.93	2.18	1.21	1.63	1.55	1.16	4.24	5.08	3.28
9	4.07	3.91	3.77	1.93	2.18	1.21	1.63	1.55	1.16	4.25	5.09	3.29
10	4.07	3.91	3.77	1.93	2.18	1.21	1.63	1.55	1.16	4.25	5.11	3.29
11	4.08	3.92	3.78	1.94	2.18	1.21	1.63	1.55	1.16	4.25	5.11	3.30
12	4.09	3.93	3.78	1.94	2.18	1.21	1.63	1.55	1.16	4.27	5.12	3.30
13	4.09	3.93	3.79	1.94	2.18	1.21	1.64	1.55	1.16	4.28	5.12	3.30
14	4.11	3.94	3.79	1.94	2.18	1.21	1.64	1.55	1.16	4.28	5.13	3.33
15	4.13	3.94	3.79	1.94	2.18	1.21	1.64	1.55	1.17	4.29	5.14	3.34
16	4.15	3.95	3.82	1.94	2.18	1.21	1.64	1.55	1.17	4.29	5.14	3.34
17	4.15	3.96	3.83	1.94	2.18	1.21	1.64	1.55	1.17	4.29	5.14	3.35
18	4.16	3.98	3.84	1.94	2.18	1.21	1.64	1.56	1.17	4.29	5.14	3.35
19	4.16	3.98	3.84	1.94	2.18	1.21	1.64	1.56	1.18	4.29	5.14	3.36
20	4.16	3.98	3.84	1.94	2.18	1.21	1.64	1.56	1.18	4.29	5.17	3.36
21	4.17	3.98	3.84	1.94	2.18	1.21	1.64	1.56	1.18	4.30	5.17	3.36
22	4.17	3.99	3.85	1.94	2.18	1.21	1.64	1.56	1.18	4.31	5.18	3.36
23	4.18	3.99	3.85	1.94	2.18	1.21	1.64	1.56	1.18	4.31	5.18	3.36
24	4.18	3.99	3.85	1.94	2.18	1.21	1.64	1.56	1.18	4.32	5.19	3.37
25	4.18	4.00	3.85	1.94	2.18	1.21	1.64	1.56	1.18	4.32	5.20	3.37
26	4.18	4.00	3.86	1.94	2.18	1.21	1.64	1.56	1.18	4.32	5.20	3.37
27	4.19	4.01	3.86	1.94	2.18	1.21	1.65	1.56	1.18	4.32	5.21	3.37
28	4.19	4.01	3.87	1.94	2.18	1.21	1.65	1.56	1.18	4.32	5.21	3.38
29	4.19	4.02	3.87	1.94	2.18	1.21	1.65	1.56	1.19	4.32	5.21	3.38
30	4.20	4.02	3.88	1.94	2.18	1.21	1.65	1.56	1.19	4.32	5.22	3.38
31	4.21	4.03	3.88	1.94	2.18	1.21	1.65	1.56	1.19	4.32	5.22	3.39
32	4.21	4.03	3.88	1.94	2.18	1.21	1.65	1.56	1.19	4.32	5.22	3.39
33	4.21	4.03	3.89	1.94	2.18	1.21	1.65	1.56	1.19	4.32	5.23	3.39
34	4.21	4.03	3.89	1.94	2.18	1.21	1.65	1.57	1.19	4.33	5.23	3.40
35	4.22	4.03	3.89	1.94	2.18	1.21	1.65	1.57	1.19	4.33	5.23	3.40
36	4.22	4.04	3.89	1.94	2.18	1.21	1.65	1.57	1.19	4.34	5.24	3.42
37	4.22	4.04	3.90	1.94	2.18	1.21	1.65	1.57	1.19	4.34	5.24	3.42
38	4.23	4.04	3.90	1.94	2.18	1.21	1.65	1.57	1.2	4.35	5.25	3.42
39	4.23	4.04	3.90	1.94	2.18	1.21	1.65	1.57	1.2	4.35	5.25	3.42
40	4.23	4.05	3.90	1.94	2.18	1.21	1.65	1.57	1.2	4.35	5.25	3.42

Gold price per cost	MORILA			SADIOLA			GEITA			CVSA		
PROBABILITY	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3	OPTION 1	OPTION 2	OPTION 3
41	4.23	4.05	3.90	1.94	2.18	1.21	1.66	1.57	1.2	4.36	5.25	3.42
42	4.23	4.05	3.91	1.94	2.18	1.21	1.66	1.57	1.2	4.36	5.25	3.42
43	4.24	4.05	3.91	1.94	2.18	1.21	1.66	1.57	1.2	4.36	5.26	3.43
44	4.24	4.05	3.92	1.94	2.18	1.21	1.66	1.57	1.21	4.37	5.26	3.43
45	4.24	4.06	3.92	1.94	2.18	1.21	1.66	1.57	1.21	4.37	5.26	3.43
46	4.25	4.06	3.92	1.94	2.18	1.21	1.66	1.57	1.21	4.37	5.27	3.43
47	4.25	4.06	3.92	1.94	2.18	1.21	1.66	1.57	1.21	4.37	5.27	3.43
48	4.25	4.06	3.92	1.94	2.18	1.21	1.66	1.57	1.21	4.38	5.27	3.44
49	4.25	4.06	3.92	1.94	2.18	1.21	1.66	1.57	1.21	4.38	5.27	3.44
50	4.26	4.07	3.93	1.94	2.18	1.21	1.66	1.58	1.21	4.39	5.28	3.44
51	4.26	4.07	3.93	1.94	2.18	1.21	1.66	1.58	1.21	4.40	5.28	3.44
52	4.28	4.07	3.93	1.94	2.18	1.21	1.67	1.58	1.21	4.40	5.28	3.45
53	4.28	4.08	3.93	1.94	2.18	1.21	1.67	1.58	1.22	4.40	5.28	3.45
54	4.29	4.08	3.94	1.94	2.18	1.21	1.67	1.58	1.22	4.40	5.28	3.45
55	4.29	4.08	3.94	1.94	2.18	1.21	1.67	1.58	1.22	4.41	5.28	3.46
56	4.29	4.08	3.94	1.94	2.19	1.21	1.67	1.58	1.22	4.41	5.29	3.46
57	4.29	4.09	3.94	1.94	2.19	1.21	1.67	1.58	1.22	4.41	5.29	3.46
58	4.29	4.10	3.95	1.94	2.19	1.21	1.67	1.58	1.22	4.41	5.29	3.46
59	4.29	4.10	3.95	1.94	2.19	1.21	1.67	1.58	1.22	4.41	5.30	3.47
60	4.29	4.10	3.96	1.94	2.19	1.21	1.67	1.58	1.23	4.42	5.31	3.47
61	4.30	4.11	3.96	1.94	2.21	1.21	1.67	1.58	1.23	4.42	5.31	3.47
62	4.31	4.12	3.97	1.94	2.21	1.21	1.68	1.58	1.23	4.42	5.31	3.47
63	4.31	4.12	3.97	1.94	2.21	1.21	1.68	1.58	1.23	4.42	5.32	3.47
64	4.31	4.12	3.97	1.94	2.21	1.21	1.68	1.58	1.23	4.42	5.32	3.47
65	4.32	4.12	3.97	1.94	2.21	1.21	1.68	1.58	1.23	4.42	5.33	3.47
66	4.32	4.12	3.97	1.94	2.21	1.21	1.68	1.58	1.23	4.43	5.33	3.48
67	4.34	4.12	3.98	1.94	2.21	1.21	1.68	1.58	1.23	4.43	5.34	3.49
68	4.34	4.12	3.98	1.94	2.21	1.21	1.68	1.58	1.23	4.43	5.34	3.49
69	4.34	4.12	3.98	1.94	2.21	1.21	1.68	1.58	1.23	4.43	5.35	3.49
70	4.34	4.13	3.98	1.94	2.21	1.21	1.68	1.58	1.24	4.43	5.35	3.49
71	4.35	4.13	3.99	1.94	2.21	1.21	1.68	1.58	1.24	4.43	5.35	3.49
72	4.35	4.13	4.00	1.94	2.21	1.21	1.68	1.58	1.24	4.43	5.36	3.50
73	4.35	4.13	4.00	1.94	2.21	1.21	1.69	1.58	1.24	4.44	5.36	3.50
74	4.35	4.14	4.01	1.94	2.21	1.21	1.69	1.59	1.24	4.44	5.36	3.51
75	4.37	4.14	4.01	1.94	2.21	1.21	1.69	1.59	1.24	4.44	5.37	3.51
76	4.38	4.15	4.01	1.94	2.21	1.21	1.69	1.59	1.24	4.45	5.37	3.51
77	4.38	4.15	4.02	1.94	2.21	1.21	1.69	1.59	1.25	4.45	5.37	3.52
78	4.38	4.15	4.02	1.94	2.21	1.21	1.69	1.59	1.25	4.45	5.38	3.52
79	4.39	4.18	4.03	1.94	2.21	1.21	1.7	1.59	1.25	4.47	5.38	3.52
80	4.39	4.18	4.03	1.94	2.21	1.21	1.7	1.59	1.25	4.47	5.38	3.52
81	4.39	4.18	4.03	1.94	2.21	1.21	1.7	1.59	1.25	4.47	5.38	3.53
82	4.40	4.19	4.04	1.94	2.21	1.21	1.7	1.59	1.25	4.47	5.39	3.53
83	4.40	4.19	4.04	1.94	2.21	1.21	1.7	1.59	1.25	4.48	5.39	3.53
84	4.40	4.19	4.04	1.94	2.21	1.22	1.7	1.6	1.26	4.48	5.39	3.54
85	4.41	4.20	4.05	1.95	2.21	1.22	1.7	1.6	1.26	4.48	5.40	3.55
86	4.41	4.20	4.05	1.95	2.22	1.22	1.71	1.6	1.27	4.49	5.42	3.55
87	4.42	4.20	4.06	1.95	2.22	1.22	1.71	1.6	1.27	4.49	5.42	3.55
88	4.42	4.22	4.06	1.95	2.22	1.22	1.71	1.6	1.27	4.50	5.43	3.55
89	4.43	4.22	4.06	1.95	2.22	1.22	1.72	1.6	1.28	4.51	5.43	3.56
90	4.44	4.22	4.07	1.95	2.22	1.22	1.72	1.62	1.28	4.52	5.43	3.57
91	4.44	4.22	4.07	1.95	2.22	1.22	1.88	1.69	1.28	4.53	5.44	3.57
92	4.44	4.22	4.09	1.95	2.22	1.22	1.93	1.72	1.28	4.53	5.45	3.58
93	4.44	4.24	4.09	1.95	2.22	1.22	1.94	1.73	1.28	4.53	5.46	3.59
94	4.46	4.24	4.10	1.95	2.22	1.22	1.98	1.74	1.29	4.54	5.48	3.59
95	4.48	4.24	4.10	1.95	2.22	1.22	1.99	1.74	1.29	4.54	5.48	3.60
96	4.49	4.24	4.11	1.95	2.22	1.22	2.01	1.74	1.29	4.56	5.48	3.60
97	4.50	4.25	4.11	1.95	2.22	1.22	2.01	1.75	1.29	4.56	5.50	3.61
98	4.52	4.30	4.16	1.95	2.22	1.22	2.03	1.75	1.29	4.57	5.51	3.64
99	4.52	4.31	4.17	1.95	2.22	1.22	2.05	1.76	1.3	4.58	5.57	3.65
99.5	4.56	4.38	4.25	1.95	2.22	1.22	2.07	1.76	1.3	4.64	5.63	3.69

Cumulative distribution Stripping ratio data for case study mines for Options 1 to 3

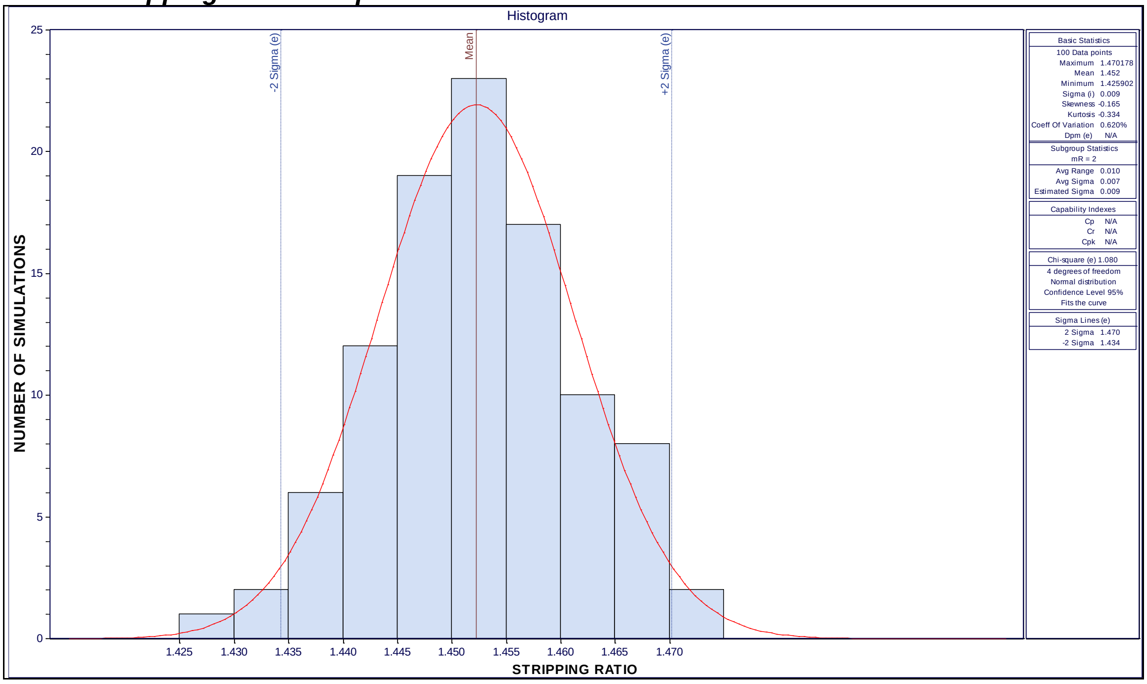
STRIPPING RATIO				
PROBABILITY	MORILA	SADIOLA	GEITA	CVSA
1	5.10	1.43	3.94	12.79
2	5.11	1.43	4.07	12.80
3	5.11	1.43	4.07	12.82
4	5.12	1.44	4.07	12.83
5	5.12	1.44	4.08	12.84
6	5.13	1.44	4.08	12.84
7	5.13	1.44	4.14	12.86
8	5.14	1.44	4.15	12.89
9	5.15	1.44	4.17	12.90
10	5.15	1.44	4.18	12.91
11	5.15	1.44	4.19	12.91
12	5.16	1.44	4.19	12.92
13	5.16	1.44	4.20	12.92
14	5.16	1.44	4.22	12.92
15	5.16	1.44	4.23	12.93
16	5.16	1.44	4.24	12.93
17	5.16	1.44	4.27	12.93
18	5.16	1.44	4.29	12.93
19	5.16	1.44	4.30	12.93
20	5.16	1.44	4.30	12.94
21	5.16	1.44	4.30	12.94
22	5.16	1.45	4.32	12.95
23	5.17	1.45	4.32	12.95
24	5.17	1.45	4.34	12.95
25	5.17	1.45	4.34	12.96
26	5.17	1.45	4.34	12.96
27	5.17	1.45	4.35	12.96
28	5.17	1.45	4.35	12.97
29	5.17	1.45	4.36	12.97
30	5.17	1.45	4.39	12.97
31	5.17	1.45	4.39	12.98
32	5.17	1.45	4.40	12.98
33	5.18	1.45	4.42	12.98
34	5.18	1.45	4.43	12.98
35	5.18	1.45	4.47	12.99
36	5.18	1.45	4.48	12.99
37	5.18	1.45	4.48	13.00
38	5.18	1.45	4.48	13.00
39	5.18	1.45	4.49	13.00
40	5.19	1.45	4.49	13.00

STRIPPING RATIO				
PROBABILITY	MORILA	SADIOLA	GEITA	CVSA
41	5.19	1.45	4.50	13.01
42	5.19	1.45	4.51	13.02
43	5.19	1.45	4.51	13.02
44	5.19	1.45	4.51	13.02
45	5.19	1.45	4.51	13.02
46	5.19	1.45	4.53	13.02
47	5.20	1.45	4.54	13.03
48	5.20	1.45	4.54	13.03
49	5.20	1.45	4.54	13.04
50	5.20	1.45	4.54	13.04
51	5.20	1.45	4.55	13.04
52	5.20	1.45	4.55	13.05
53	5.20	1.45	4.57	13.05
54	5.20	1.45	4.57	13.05
55	5.20	1.45	4.57	13.06
56	5.20	1.45	4.58	13.07
57	5.20	1.45	4.58	13.08
58	5.20	1.45	4.59	13.08
59	5.20	1.45	4.62	13.08
60	5.20	1.45	4.62	13.09
61	5.21	1.45	4.63	13.09
62	5.21	1.45	4.66	13.09
63	5.21	1.45	4.66	13.10
64	5.21	1.46	4.66	13.11
65	5.21	1.46	4.67	13.11
66	5.21	1.46	4.68	13.12
67	5.22	1.46	4.68	13.12
68	5.22	1.46	4.71	13.12
69	5.22	1.46	4.72	13.13
70	5.22	1.46	4.74	13.13
71	5.22	1.46	4.75	13.14
72	5.22	1.46	4.75	13.14
73	5.22	1.46	4.76	13.14
74	5.23	1.46	4.79	13.14
75	5.23	1.46	4.81	13.14
76	5.23	1.46	4.81	13.15
77	5.23	1.46	4.82	13.15
78	5.23	1.46	4.83	13.15
79	5.23	1.46	4.88	13.15
80	5.23	1.46	4.88	13.17
81	5.23	1.46	4.88	13.17
82	5.23	1.46	4.92	13.17
83	5.24	1.46	4.93	13.18

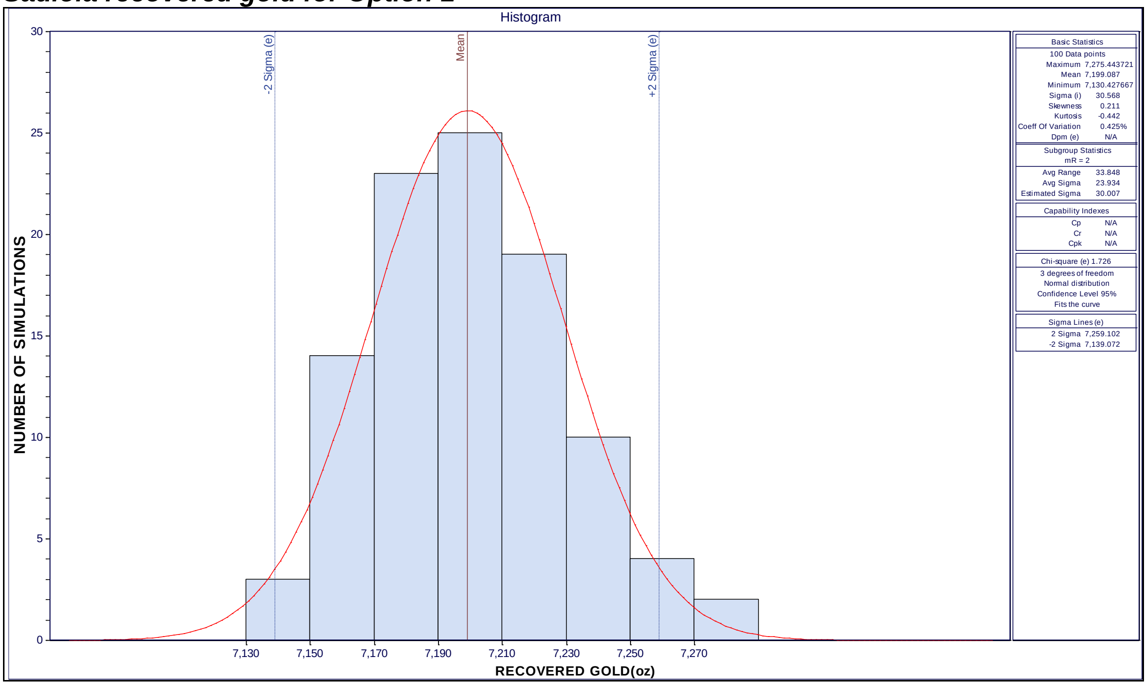
STRIPPING RATIO				
PROBABILITY	MORILA	SADIOLA	GEITA	CVSA
84	5.24	1.46	4.94	13.19
85	5.24	1.46	4.97	13.19
86	5.24	1.46	4.99	13.19
87	5.24	1.46	5.01	13.20
88	5.24	1.46	5.03	13.21
89	5.24	1.46	5.05	13.22
90	5.25	1.46	5.08	13.24
91	5.25	1.47	9.91	13.26
92	5.25	1.47	10.13	13.28
93	5.25	1.47	10.48	13.30
94	5.26	1.47	10.80	13.30
95	5.27	1.47	10.82	13.31
96	5.27	1.47	10.87	13.32
97	5.28	1.47	11.02	13.35
98	5.29	1.47	11.18	13.40
99	5.30	1.47	11.28	13.44
99.5	5.33	1.47	11.85	13.46

Appendix 9: Statistical summary for Sadiola transition indicators

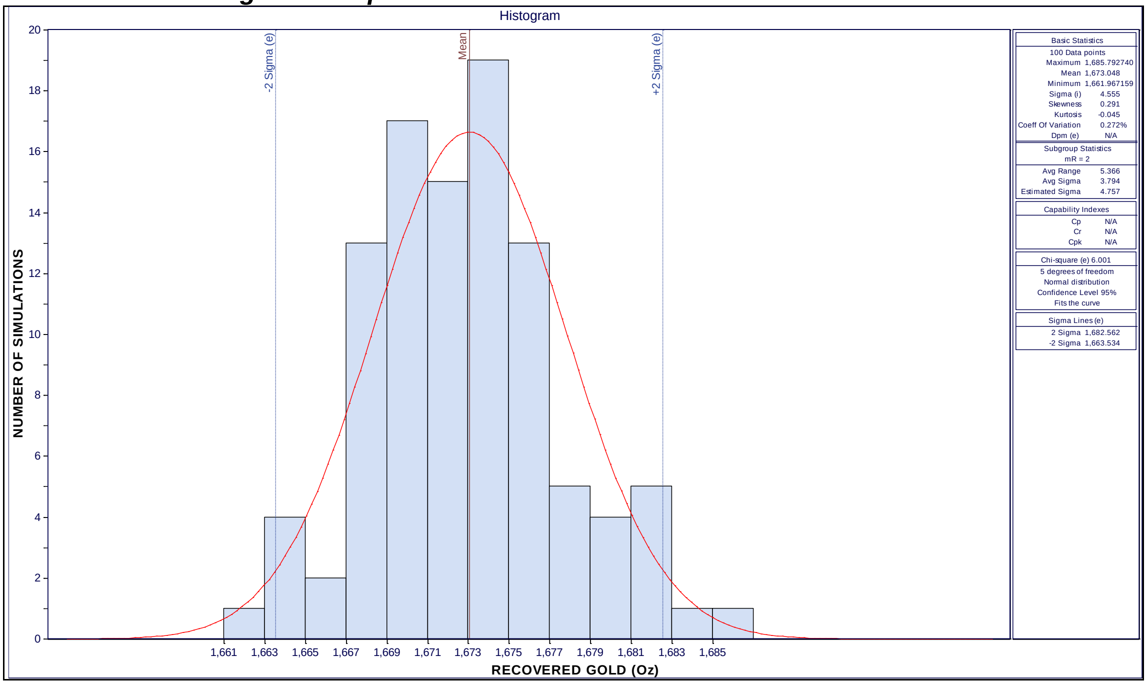
Sadiola stripping ratio for Option 1



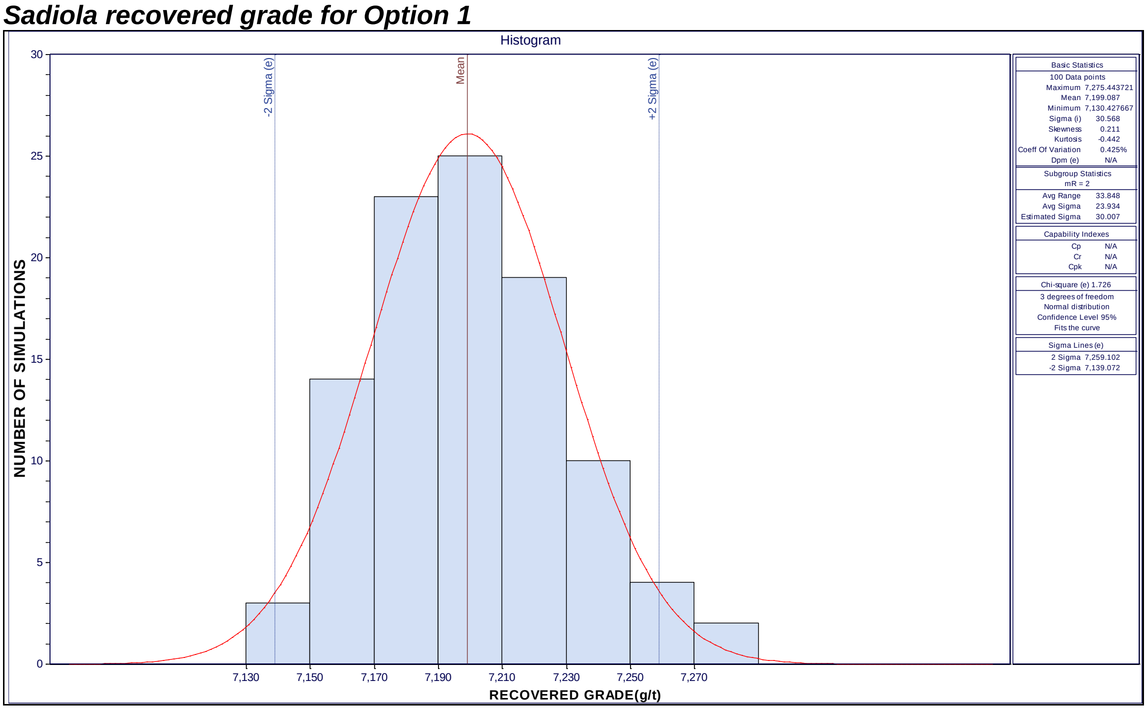
Sadiola recovered gold for Option 1



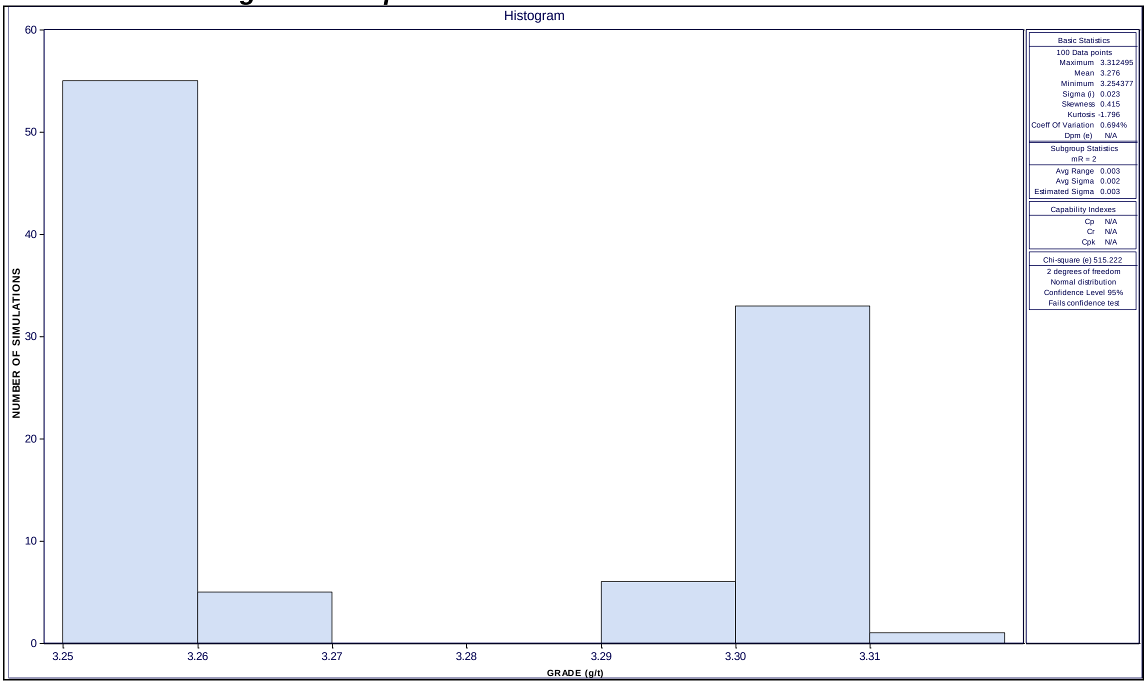
Sadiola recovered gold for Option 3



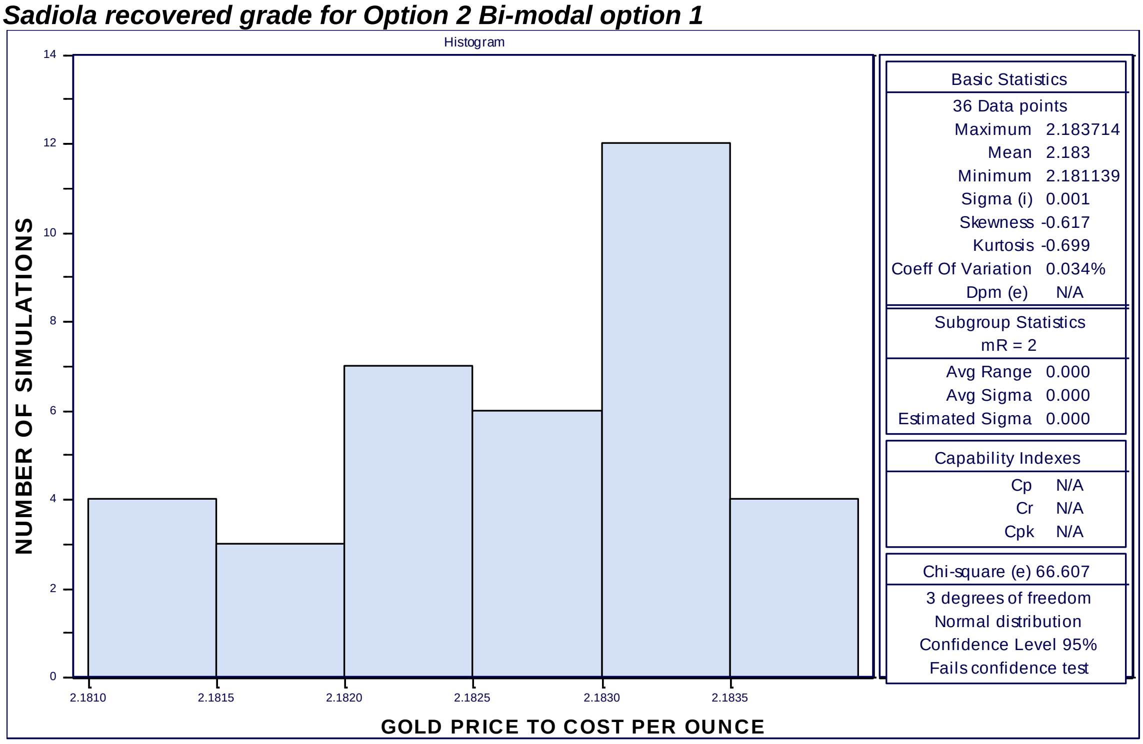
Sadiola recovered grade for Option 1



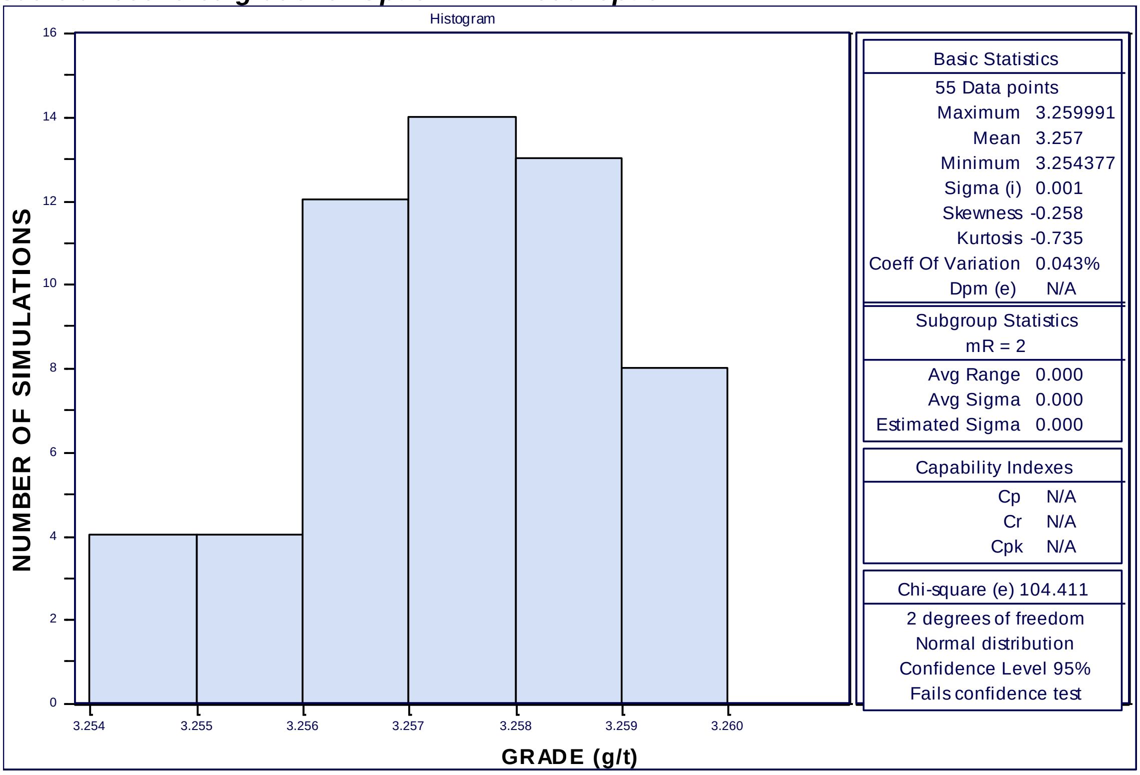
Sadiola recovered grade for Option 2



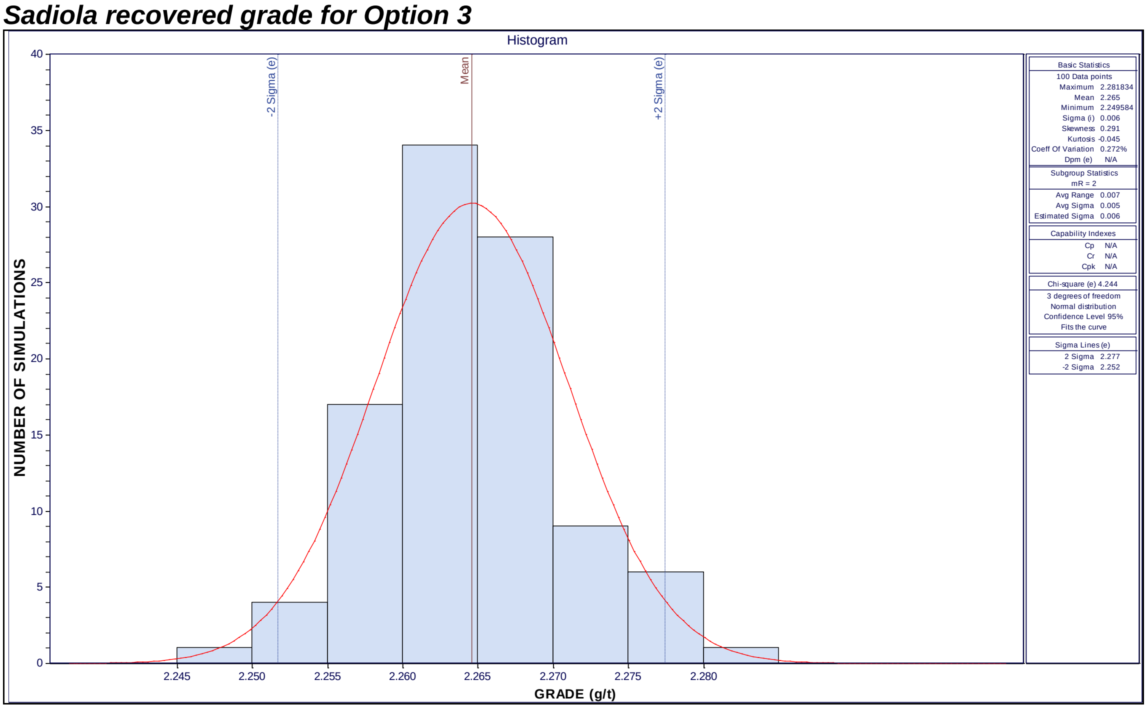
Sadiola recovered grade for Option 2 Bi-modal option 1



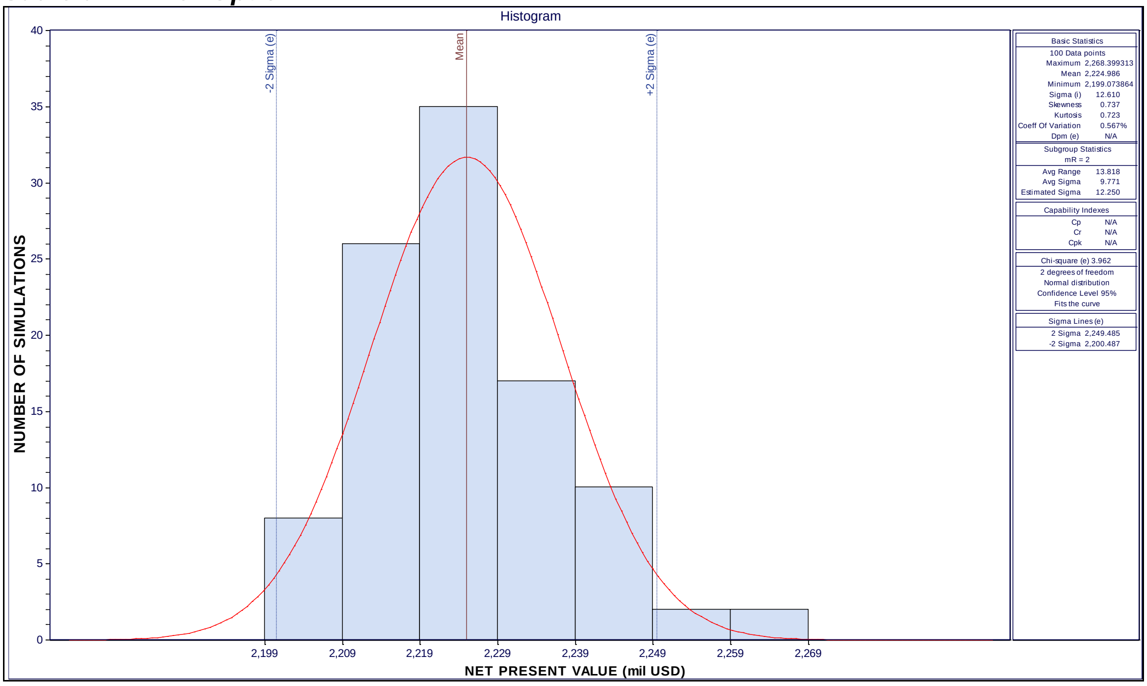
Sadiola recovered grade for Option 2 Bi-modal option 2



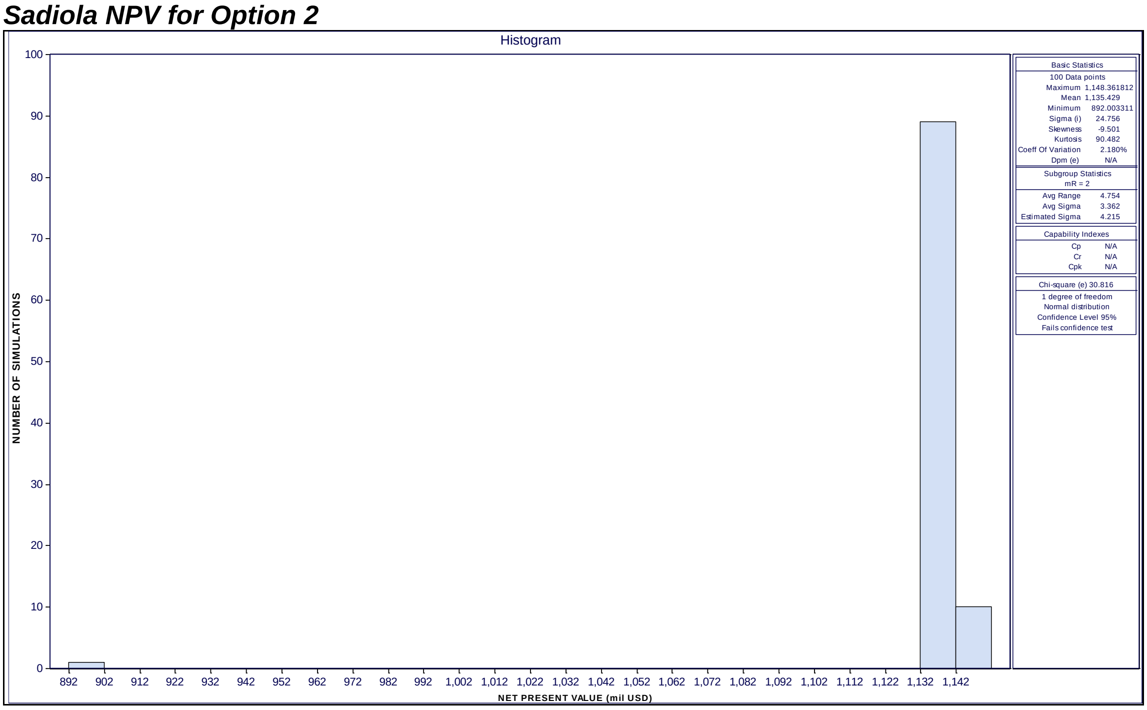
Sadiola recovered grade for Option 3



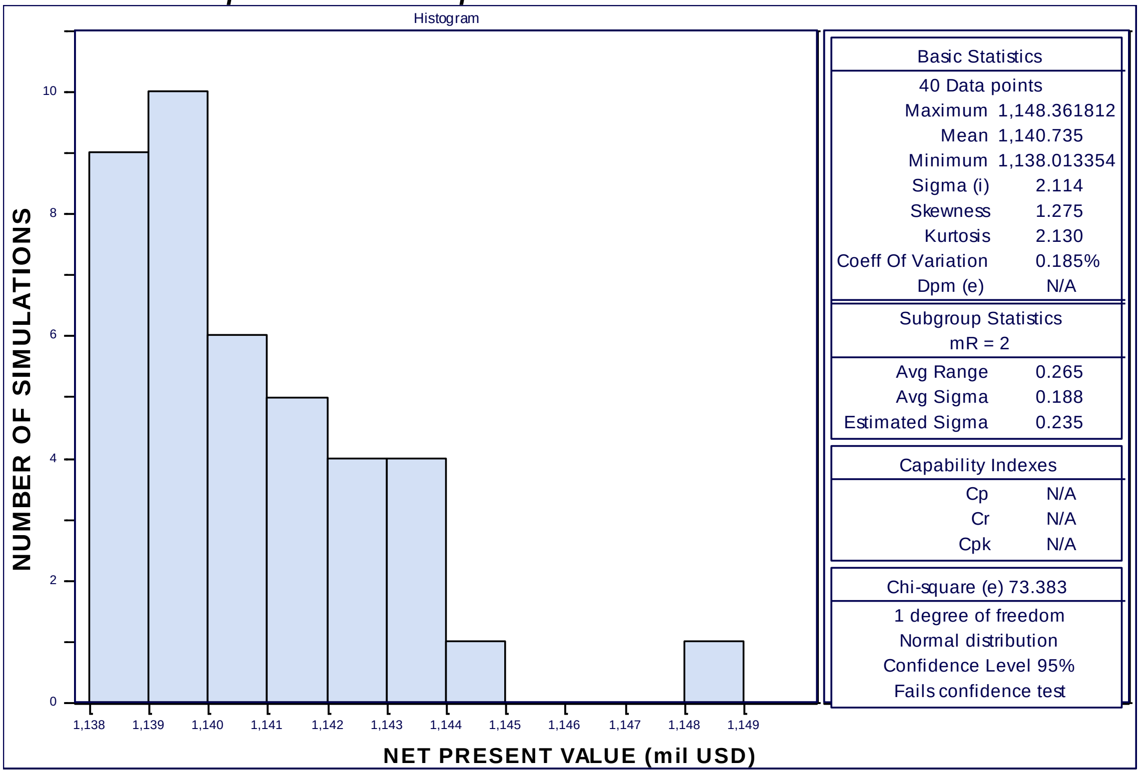
Sadiola NPV for Option 1



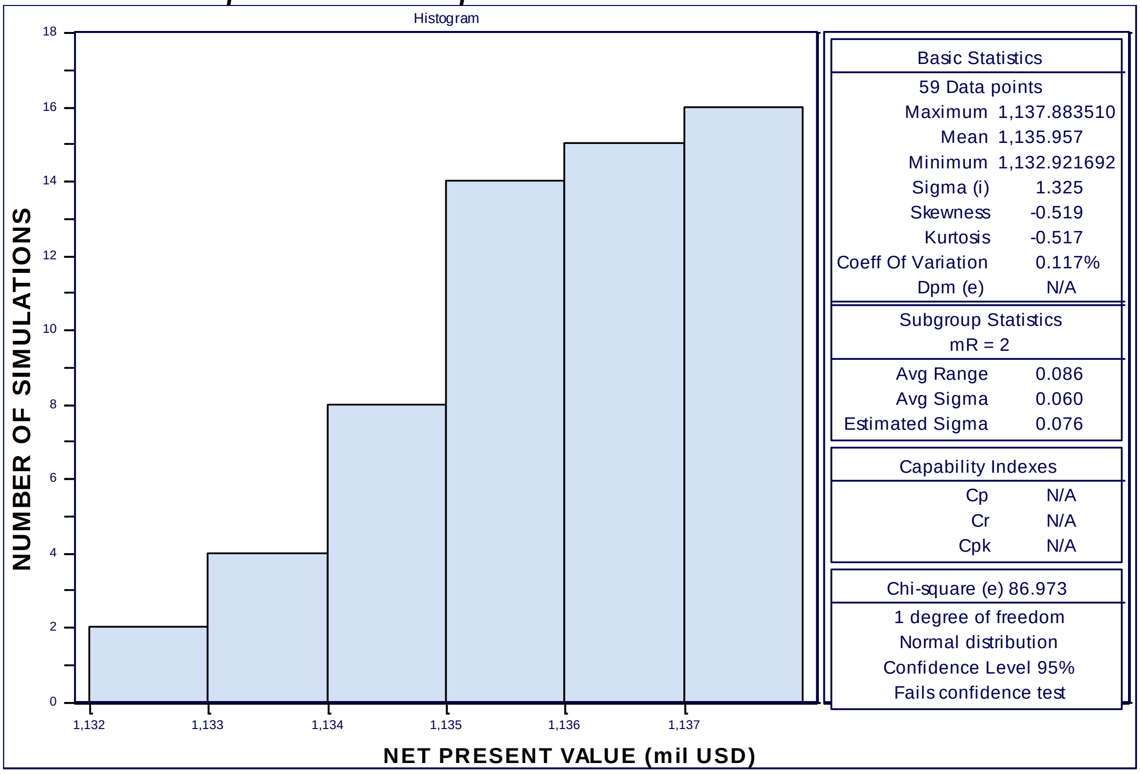
Sadiola NPV for Option 2



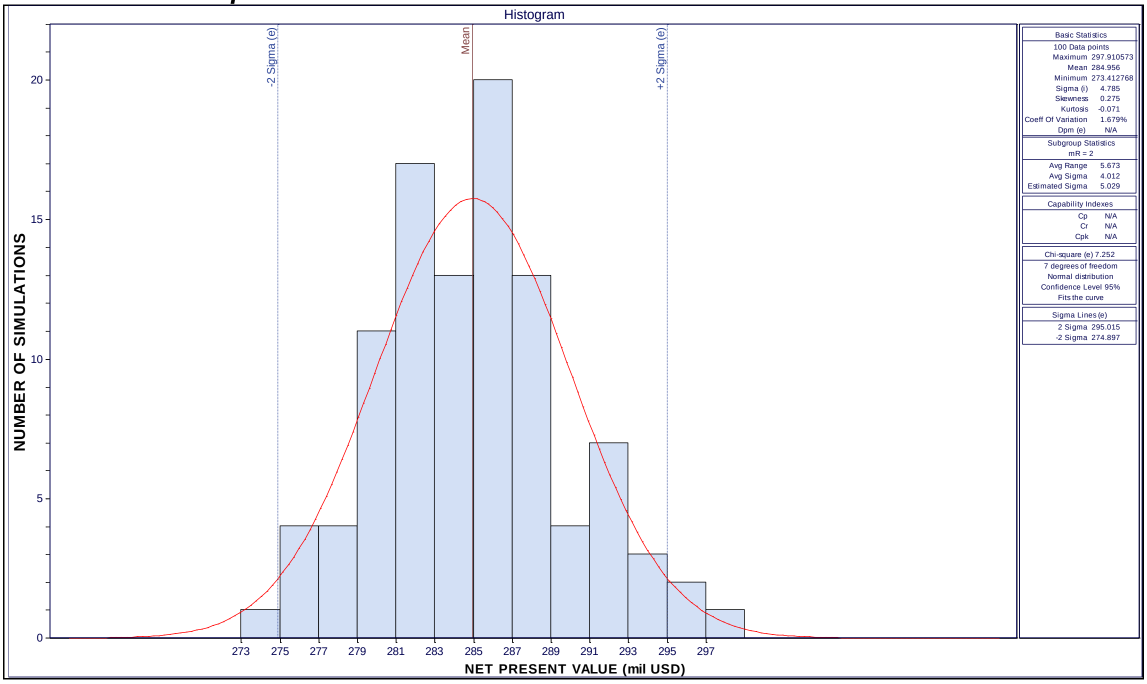
Sadiola NPV for Option 2 Bi-modal option 1



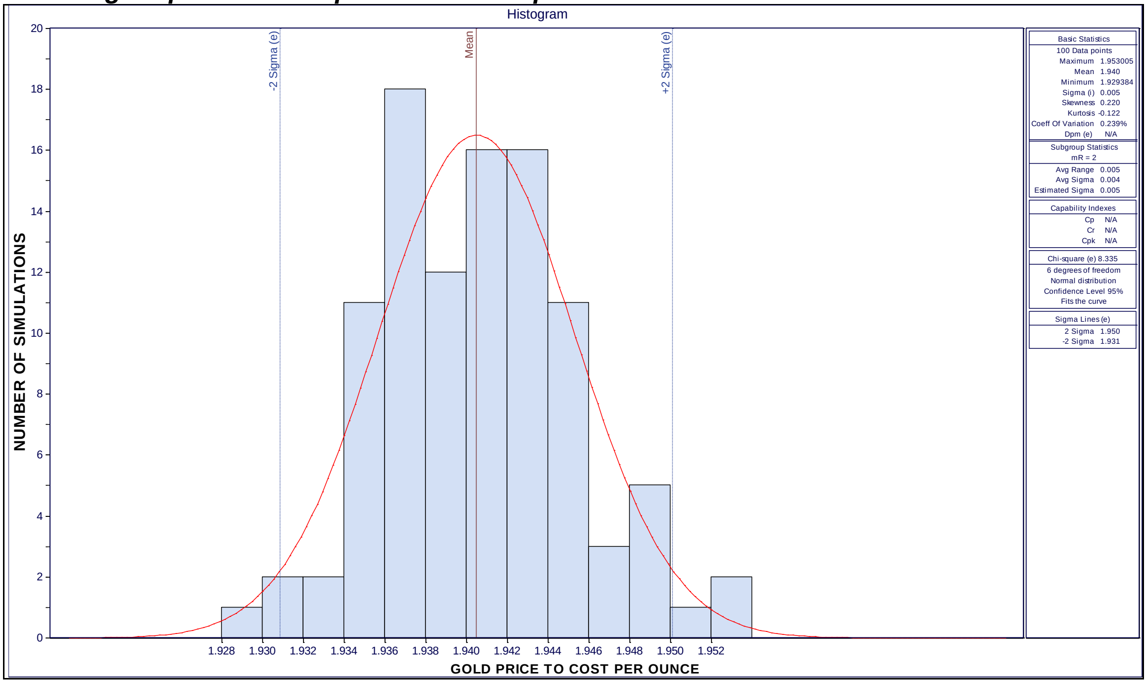
Sadiola NPV for Option 2 Bi-modal option 2



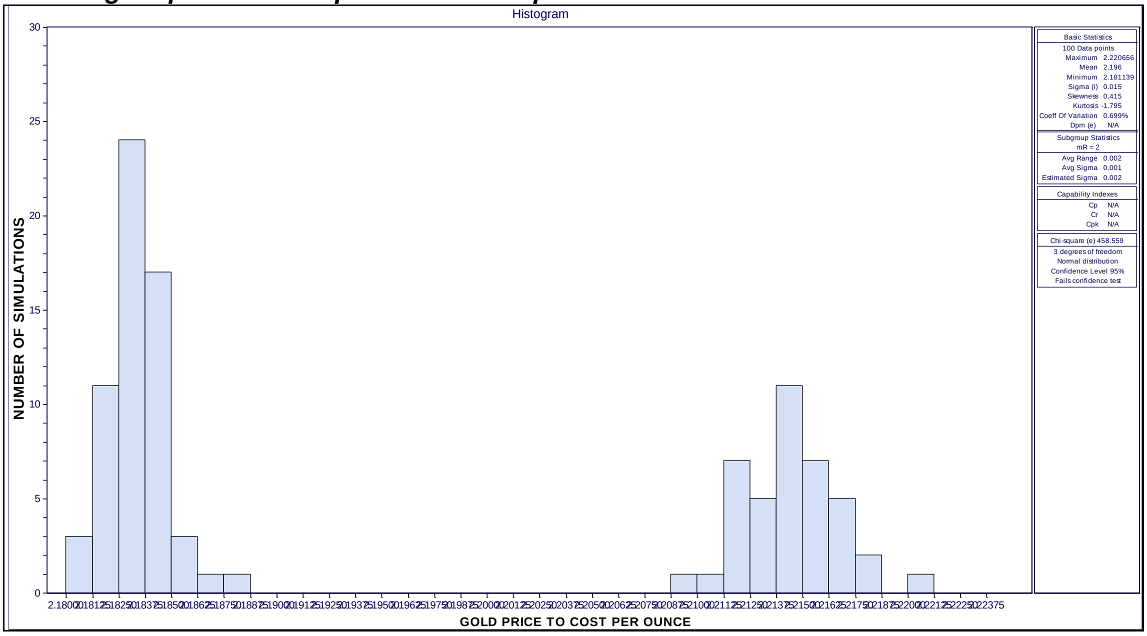
Sadiola NPV for Option 3



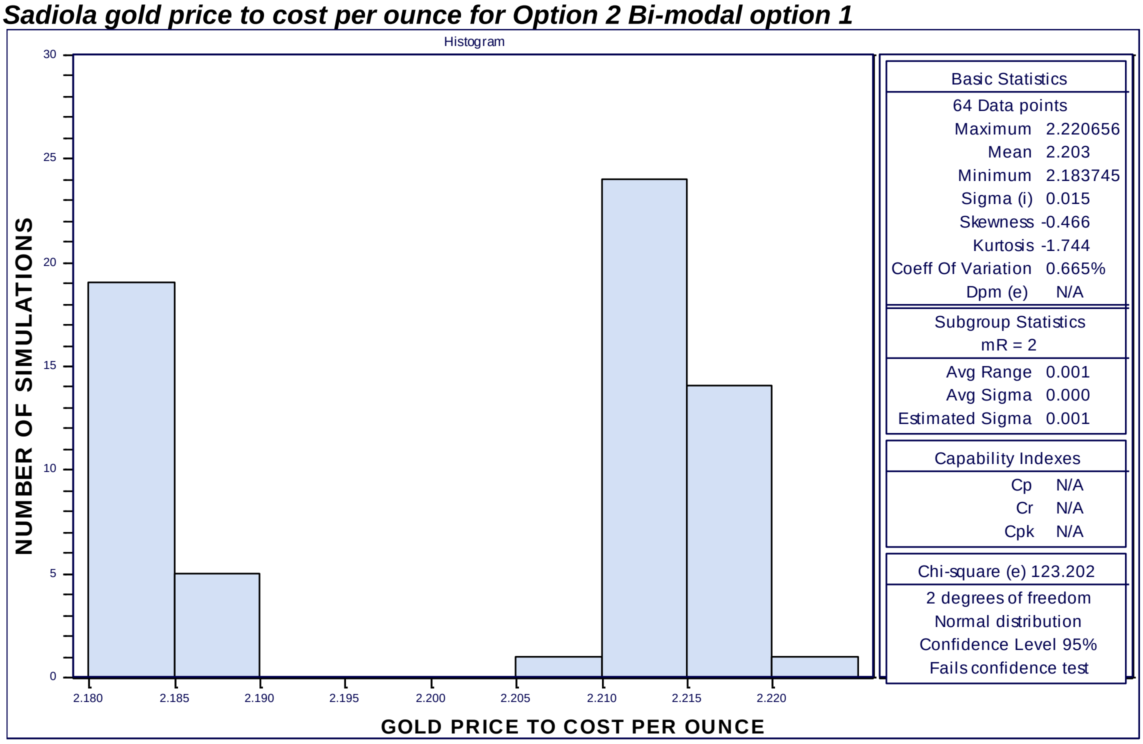
Sadiola gold price to cost per ounce for Option 1



Sadiola gold price to cost per ounce for Option 2

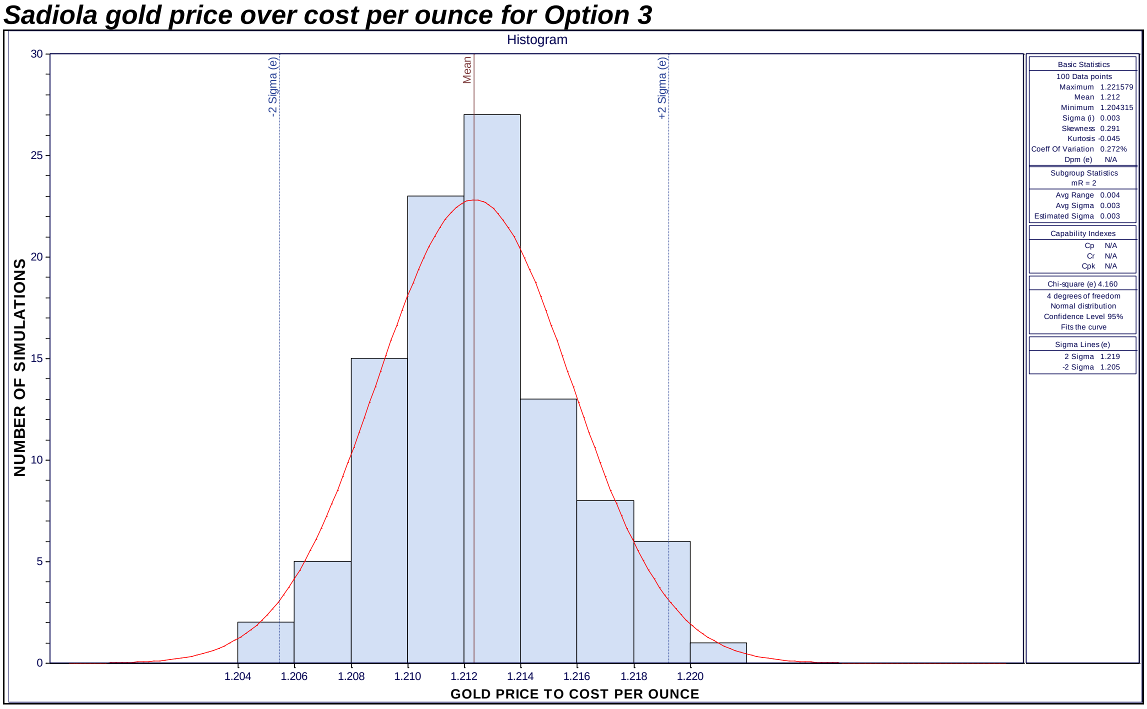


Sadiola gold price to cost per ounce for Option 2 Bi-modal option 1



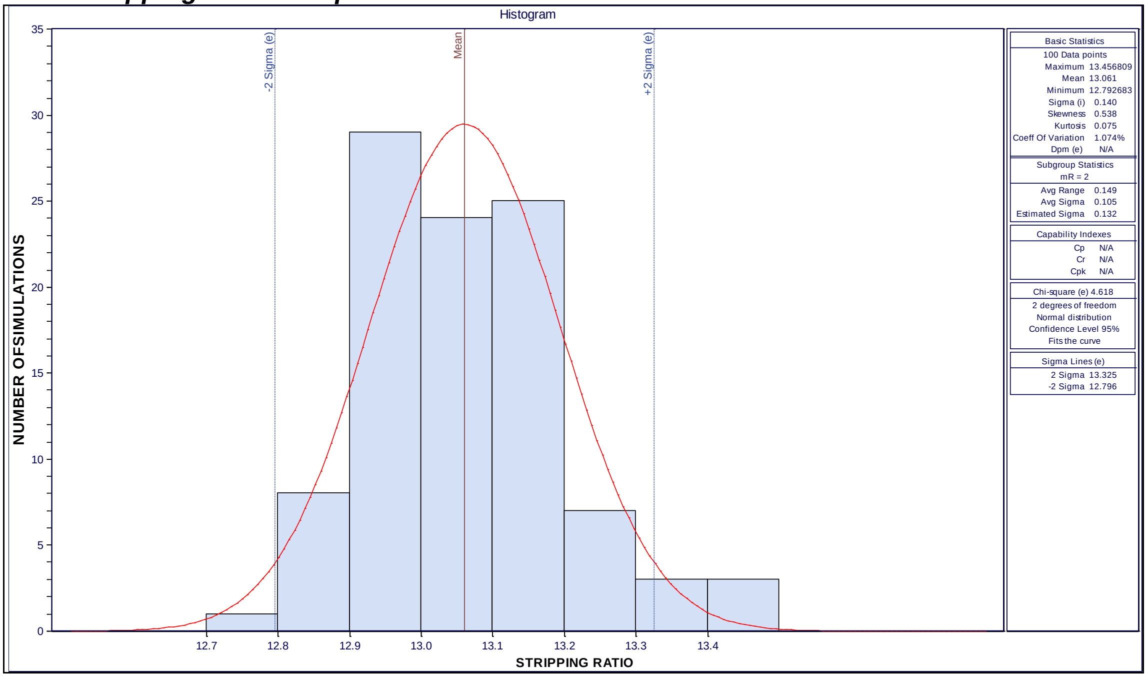
Sadiola gold price to cost per ounce for Option 2 Bi-modal option 1

Sadiola gold price over cost per ounce for Option 3

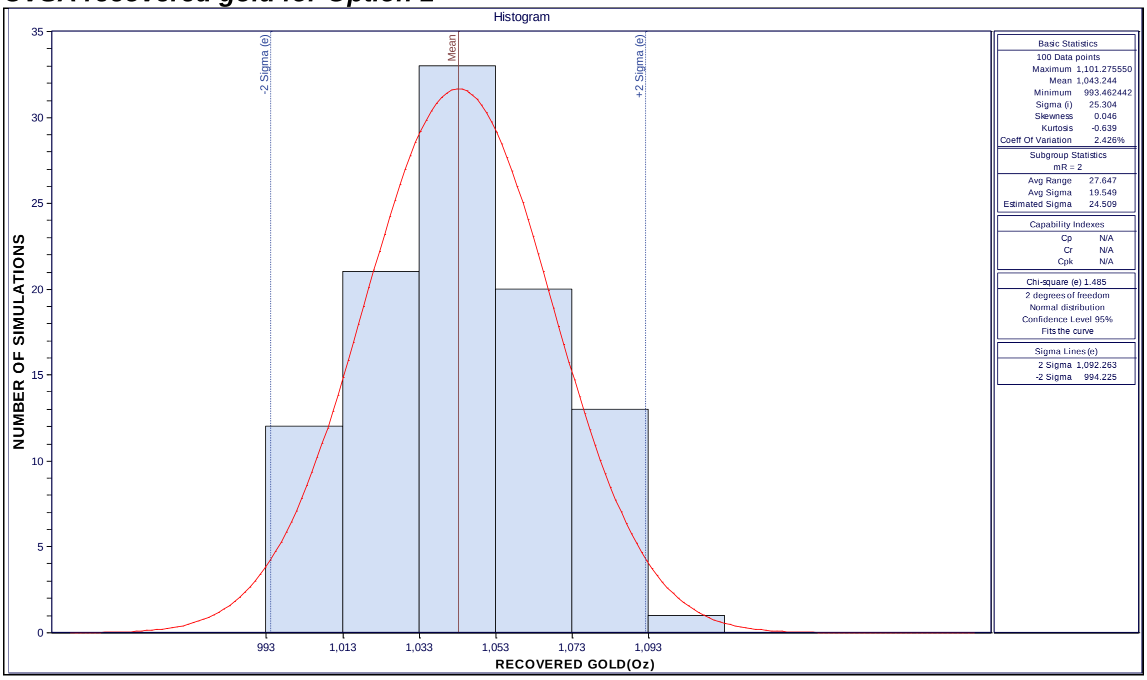


Appendix 10: Statistical summary for CVSA transition indicators

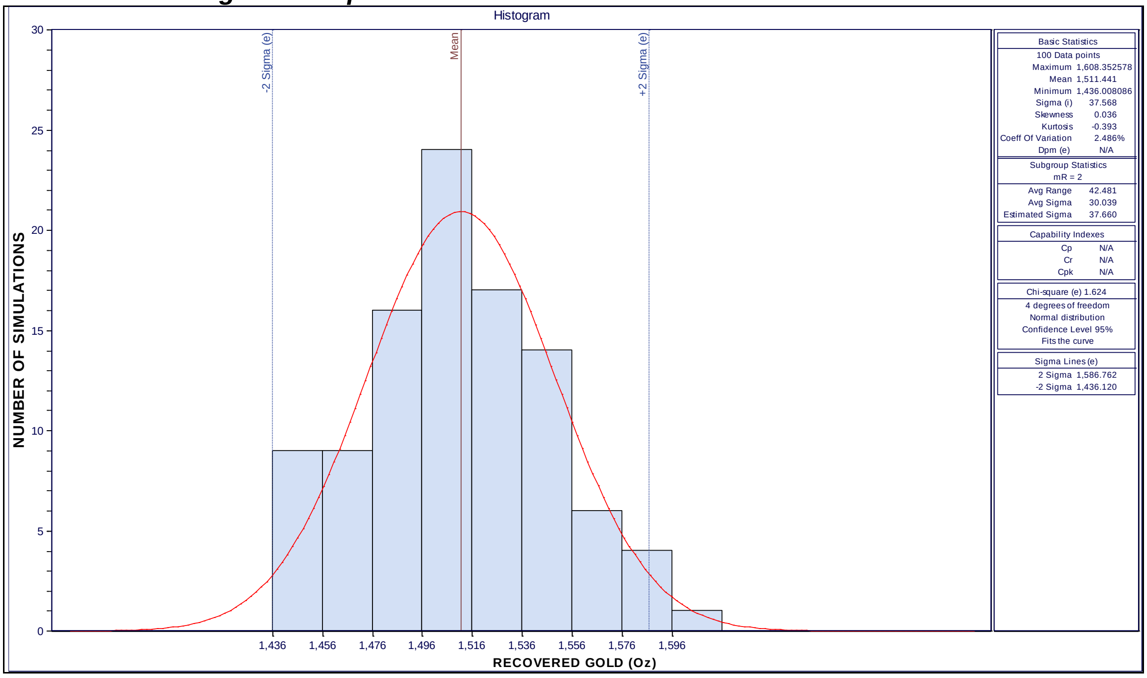
CVSA stripping ratio for Option 1



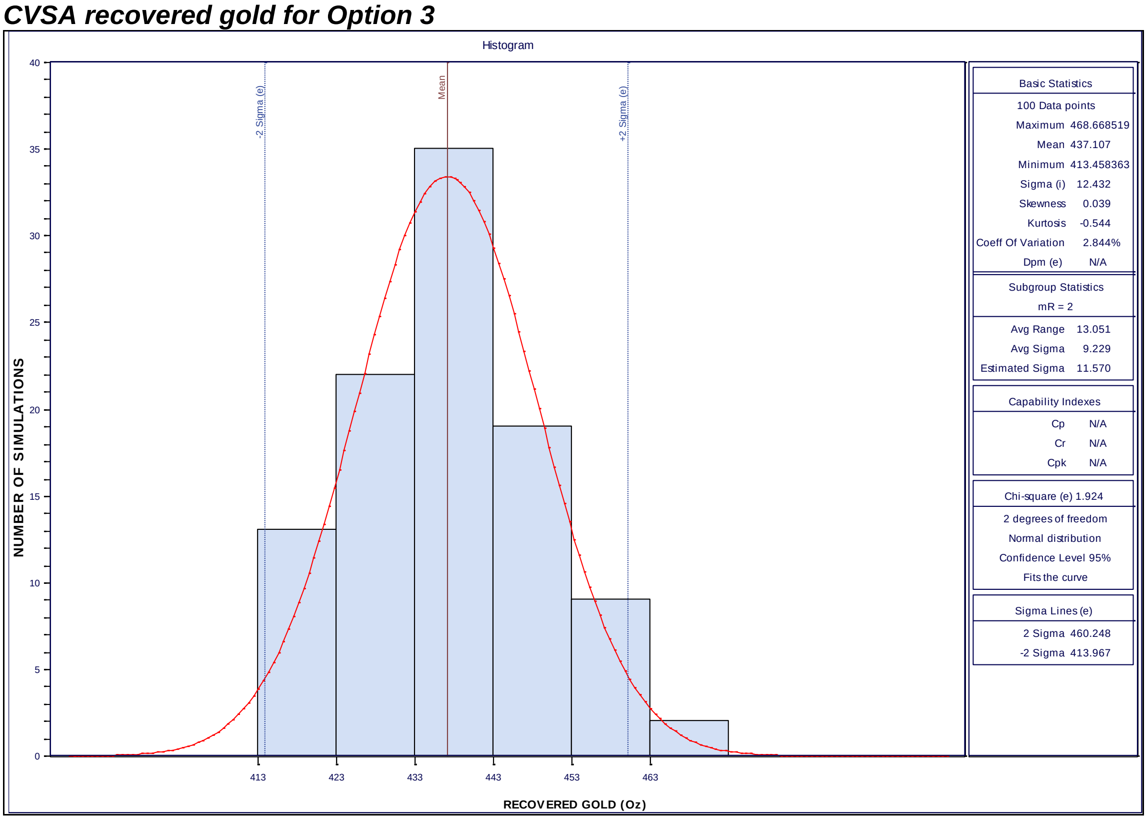
CVSA recovered gold for Option 1



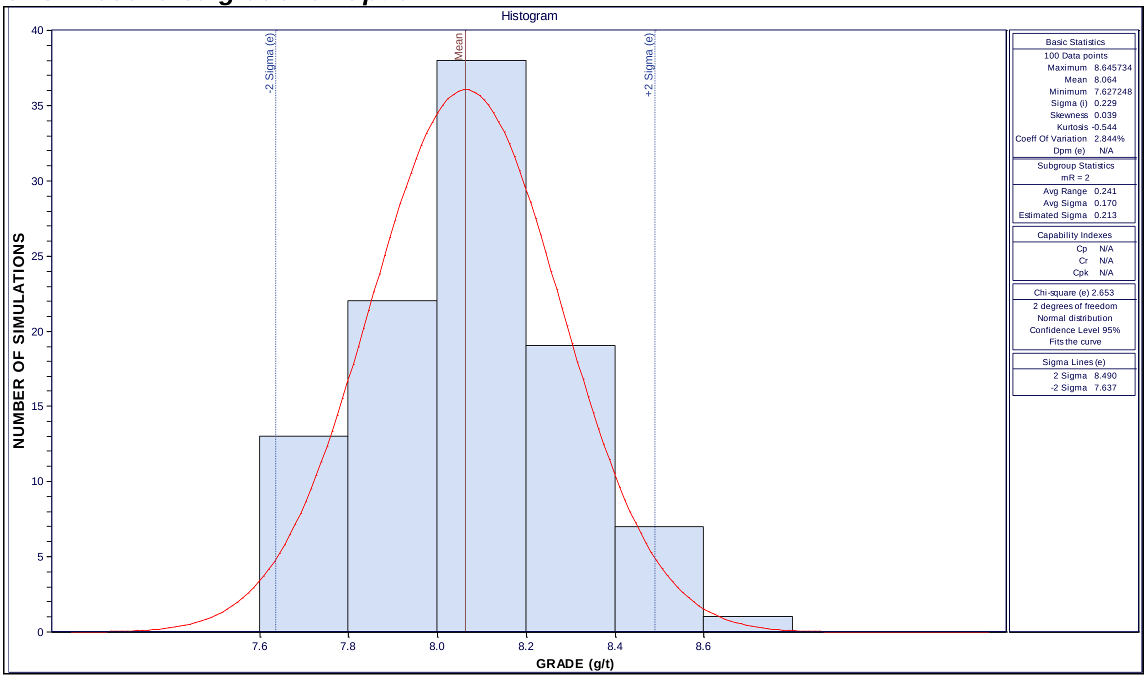
CVSA recovered gold for Option 2



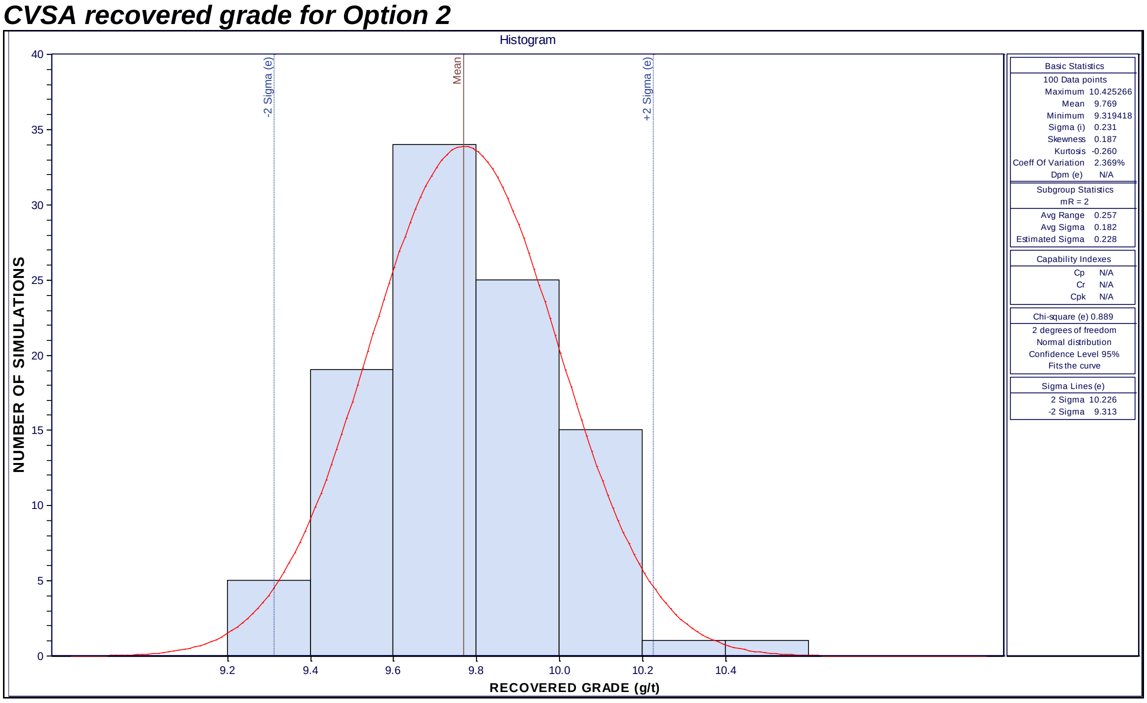
CVSA recovered gold for Option 3



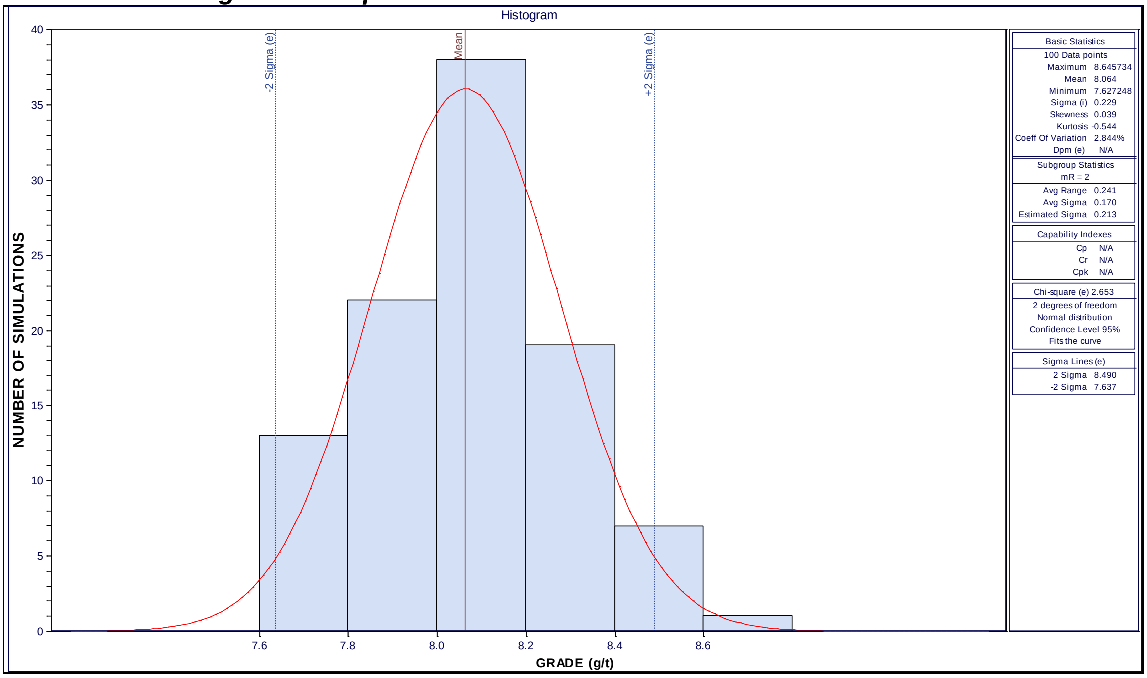
CVSA recovered grade for Option 1



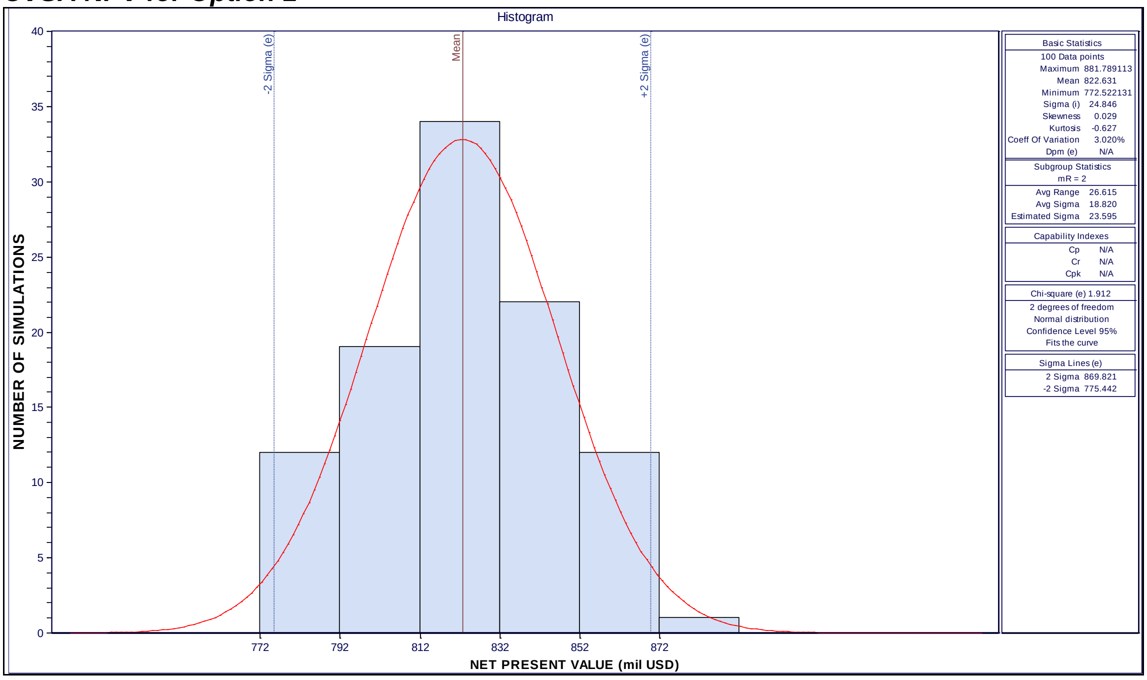
CVSA recovered grade for Option 2



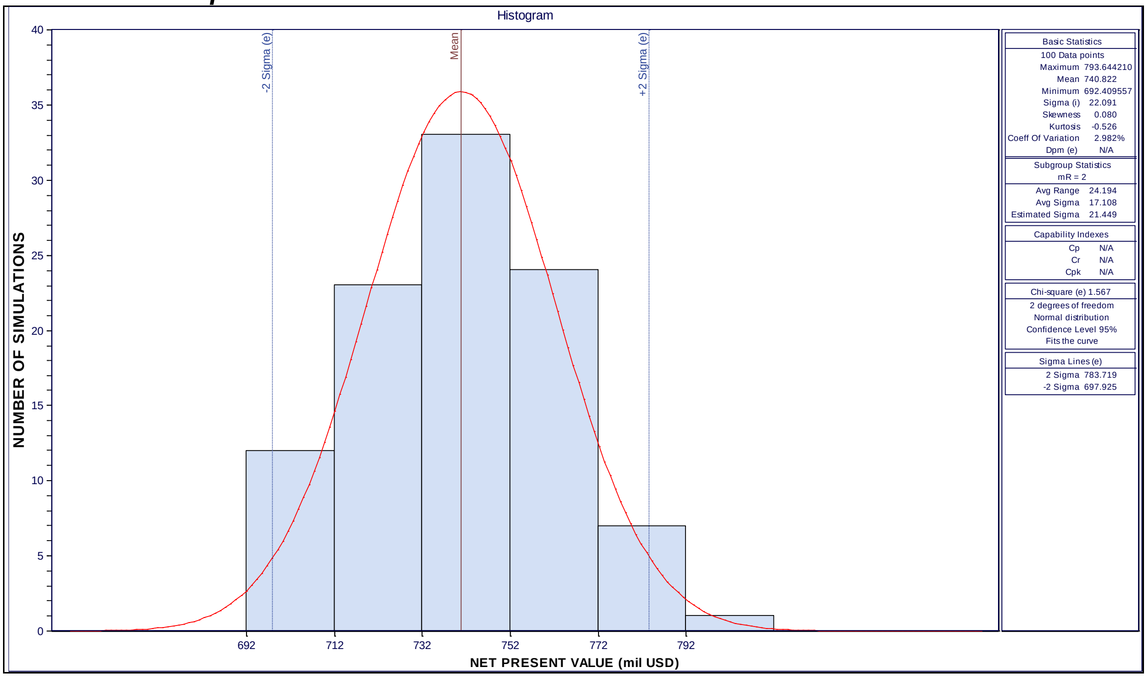
CVSA recovered grade for Option 3



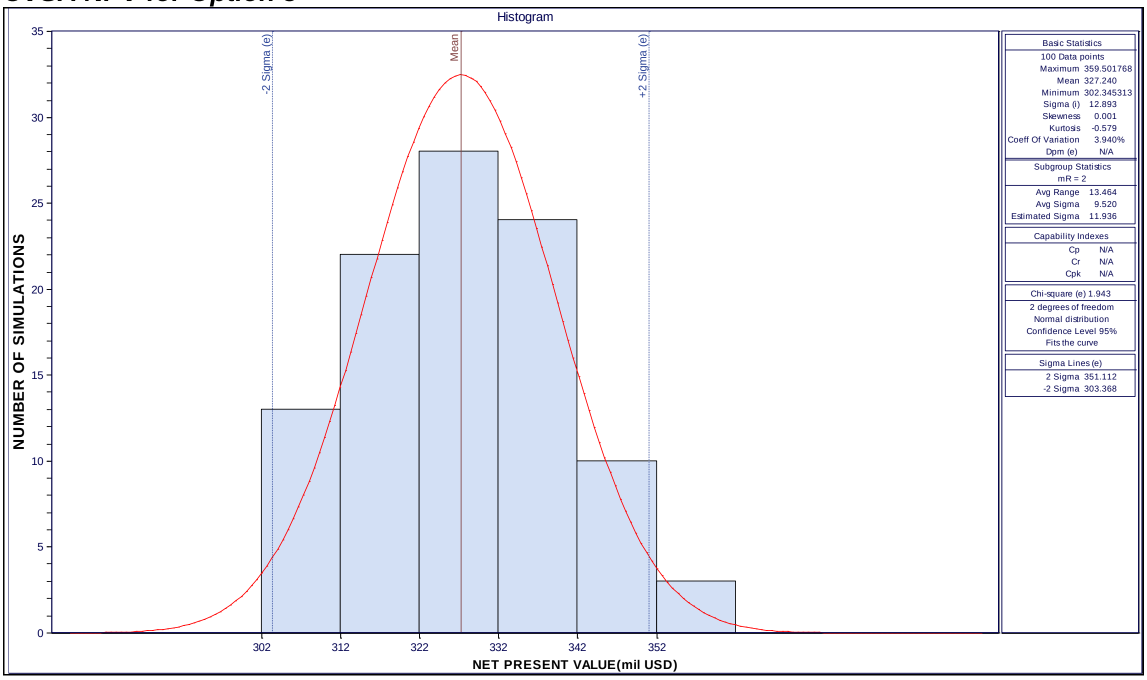
CVSA NPV for Option 1



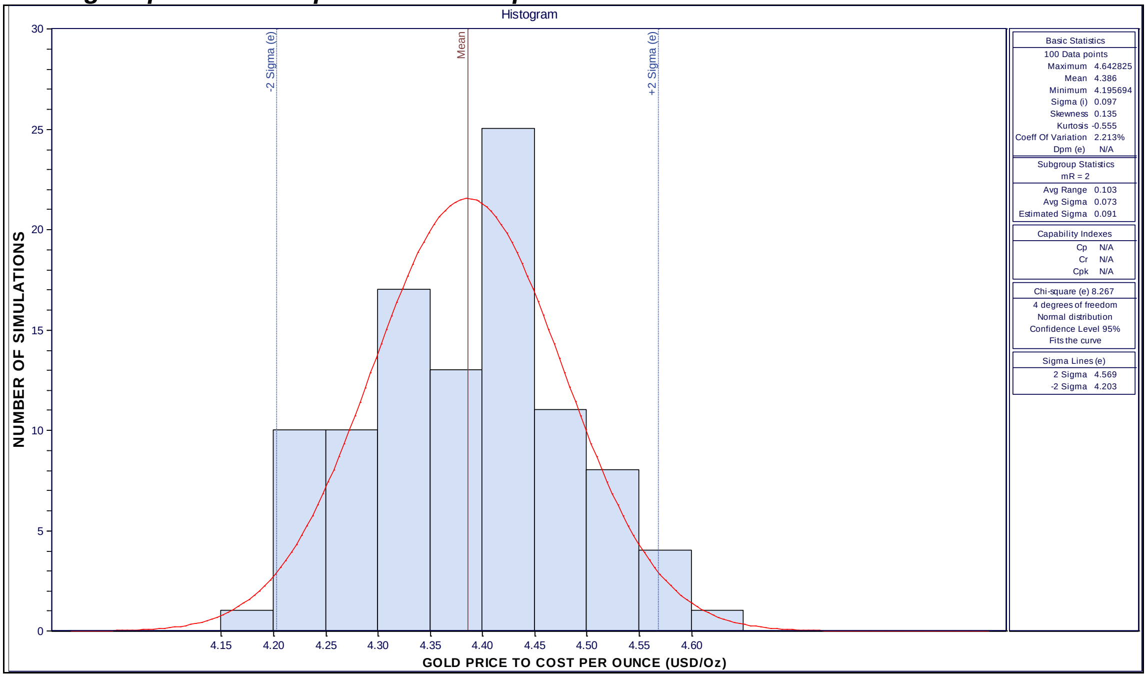
CVSA NPV for Option 2



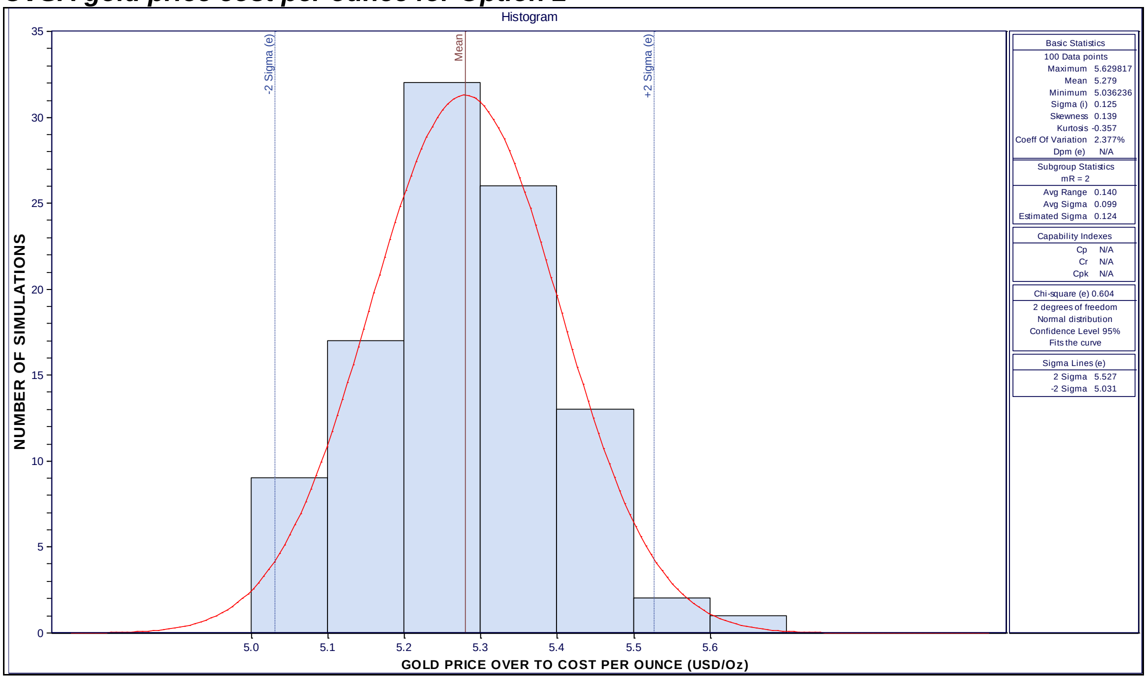
CVSA NPV for Option 3



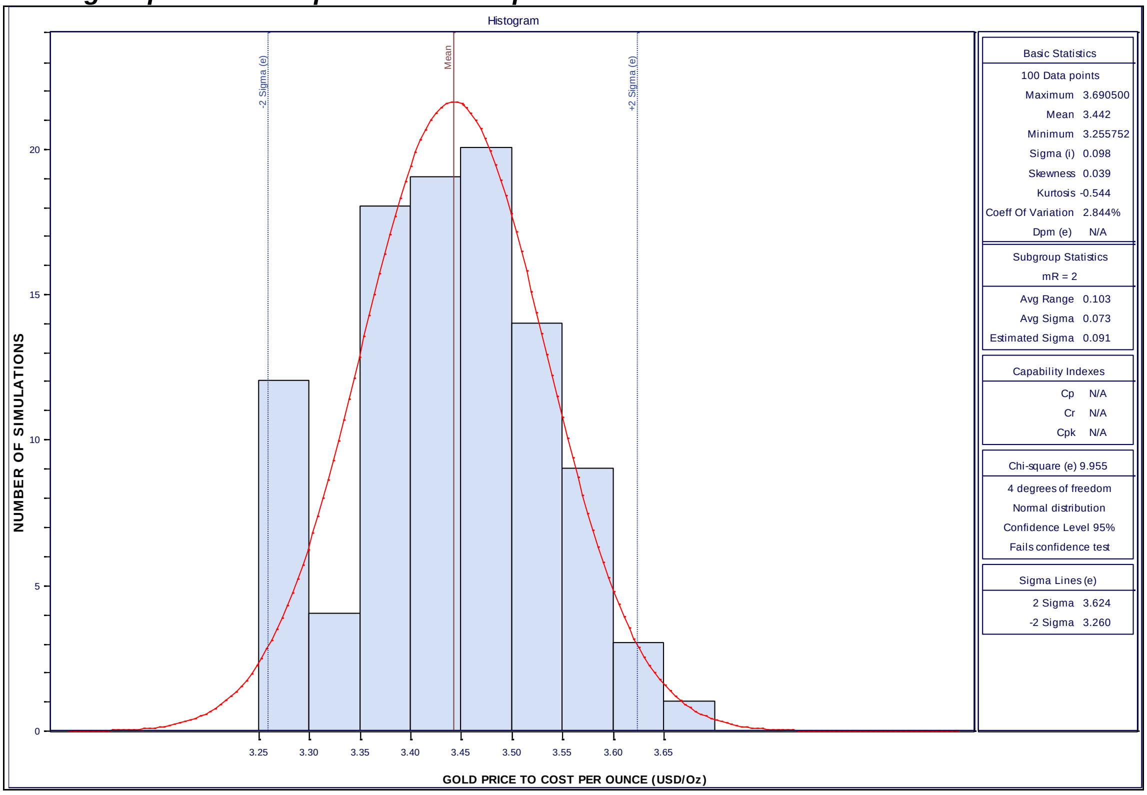
CVSA gold price to cost per ounce for Option 1



CVSA gold price cost per ounce for Option 2

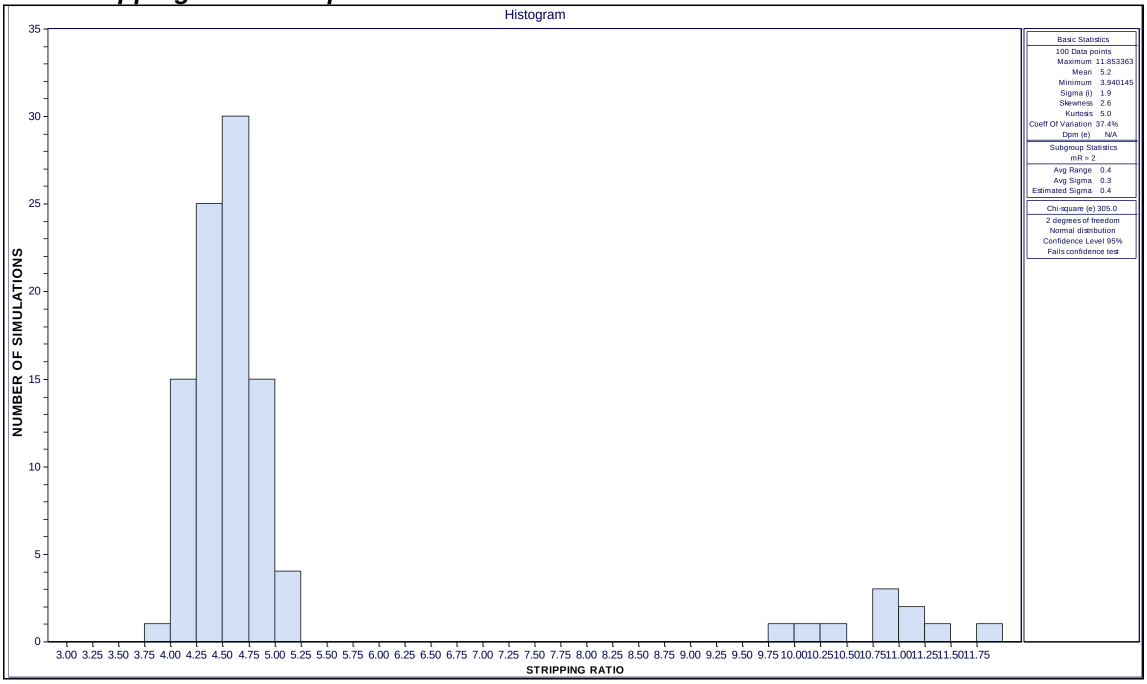


CVSA gold price to cost per ounce for Option 3

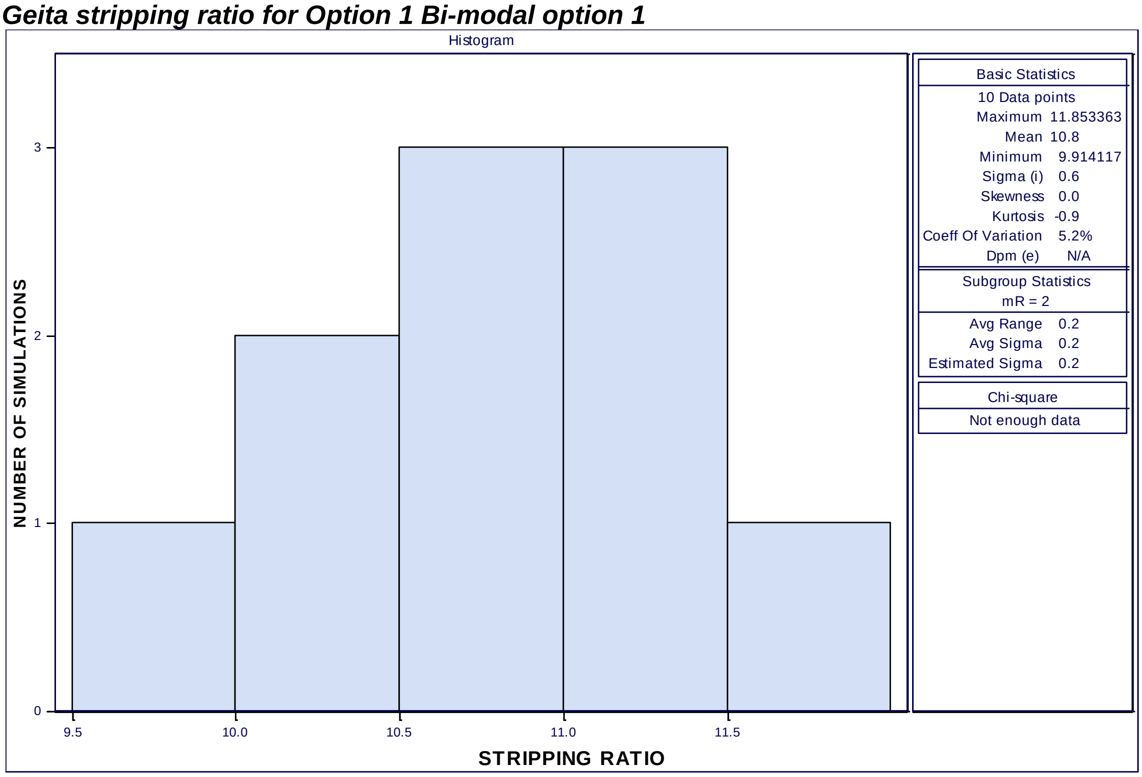


Appendix 11: Statistical summary for Geita transition indicators

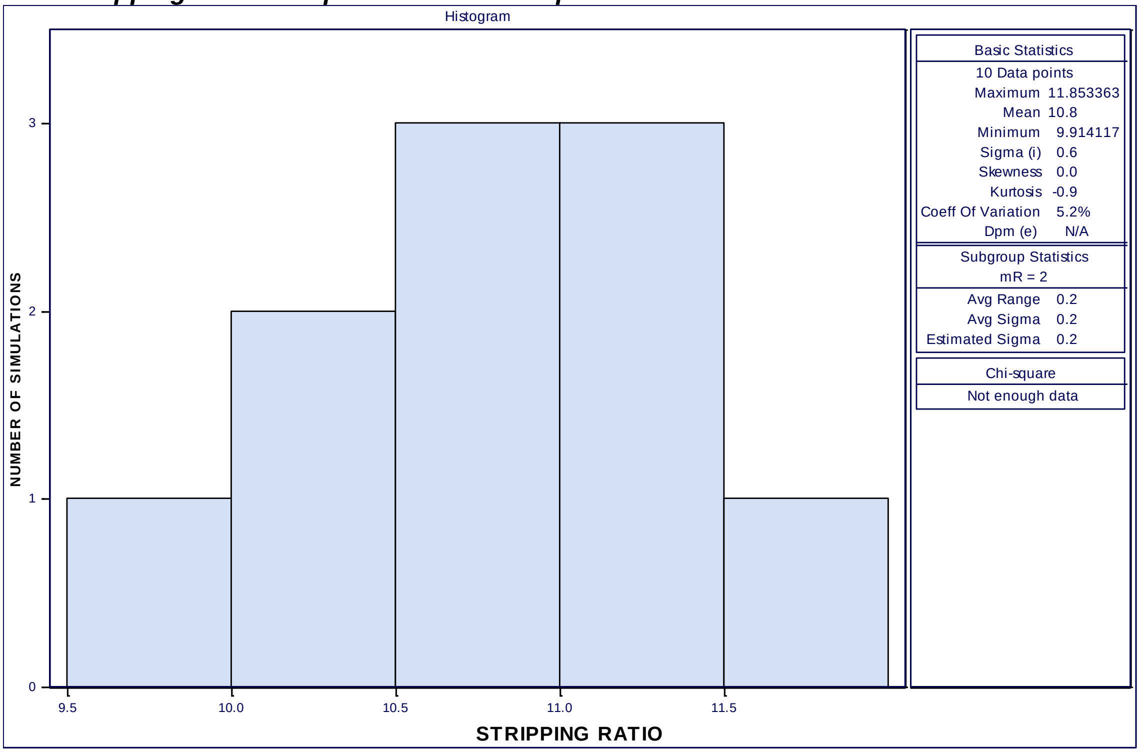
Geita stripping ratio for Option 1



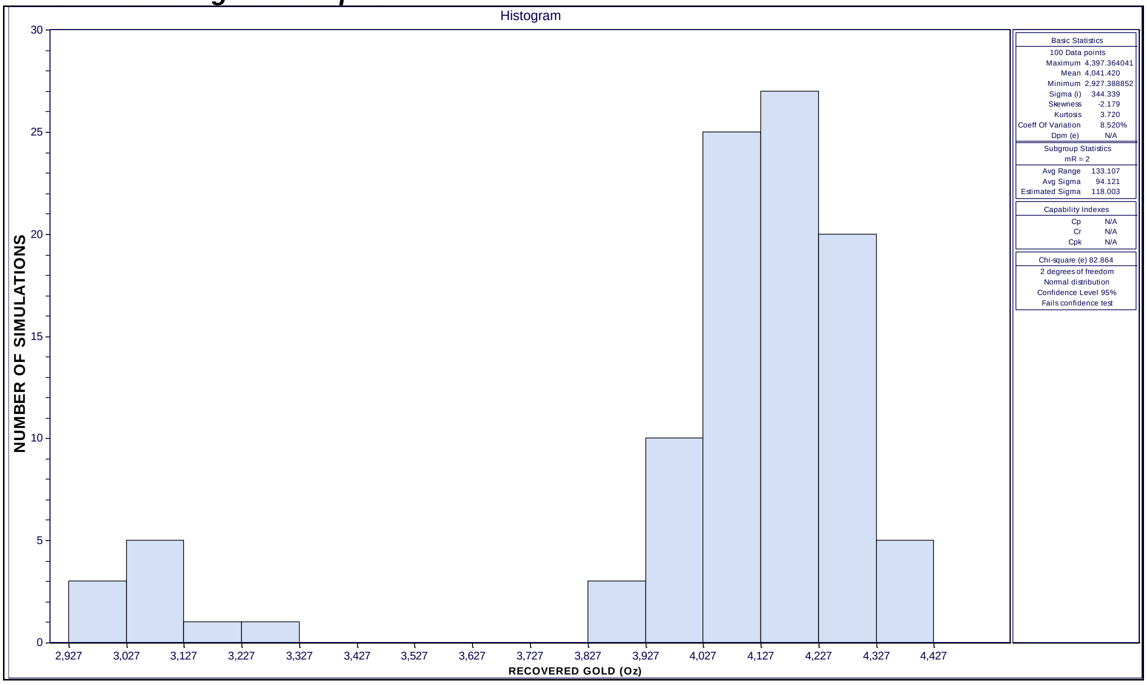
Geita stripping ratio for Option 1 Bi-modal option 1



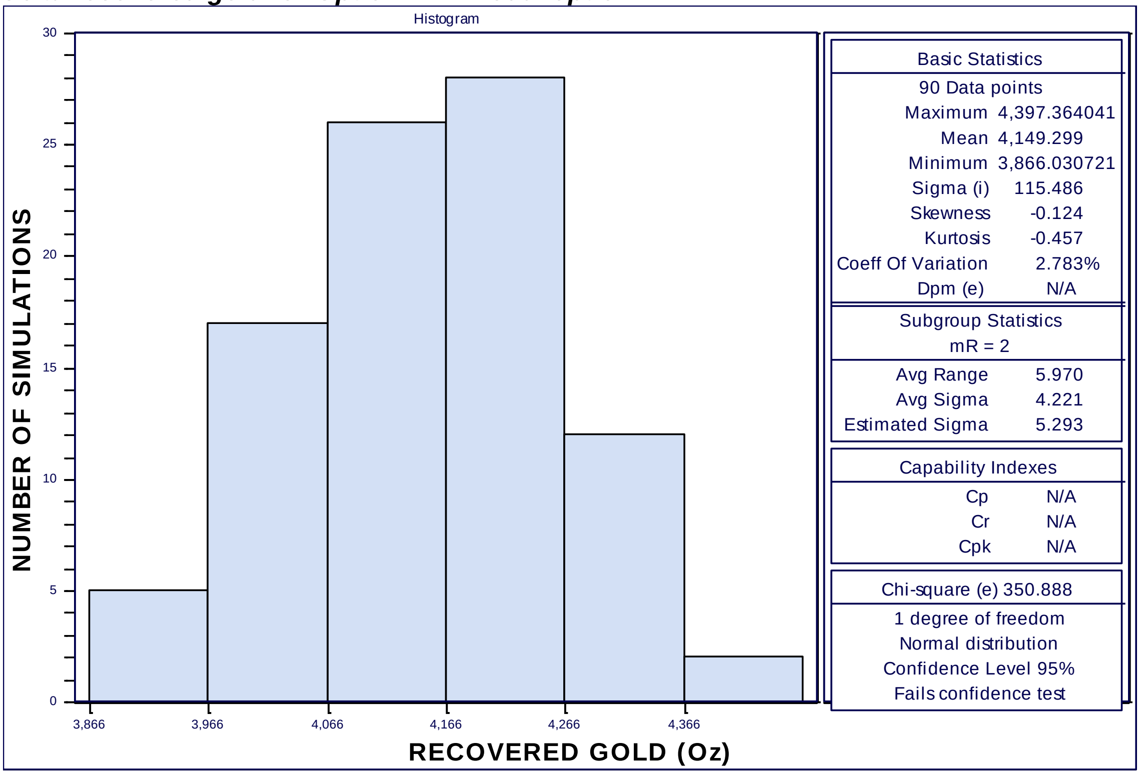
Geita stripping ratio for Option 1 Bi-modal option 2



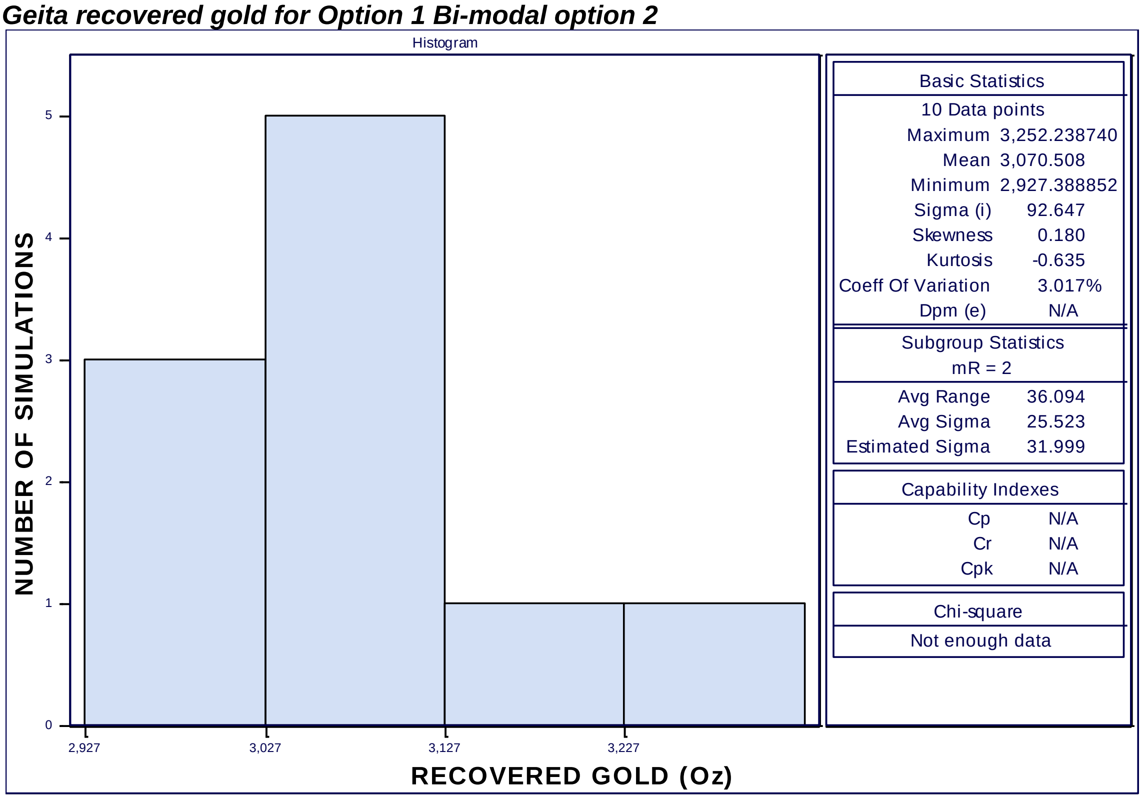
Geita recovered gold for Option 1



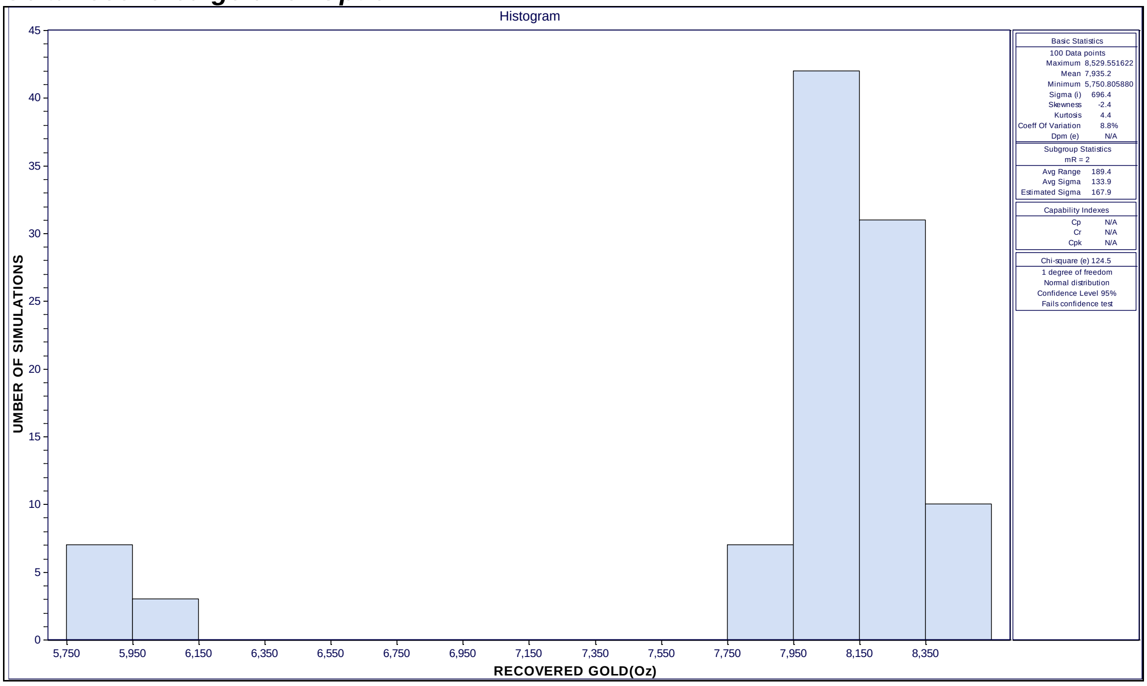
Geita recovered gold for Option 1 Bi-modal option 1



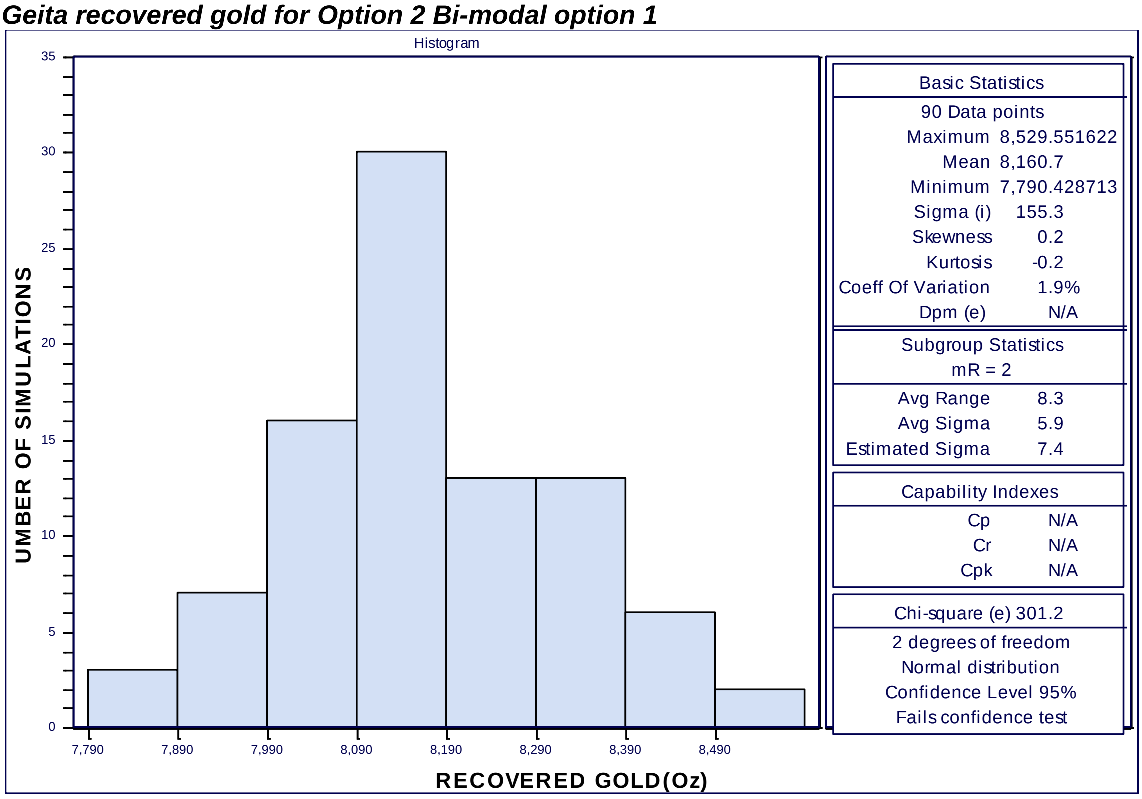
Geita recovered gold for Option 1 Bi-modal option 2



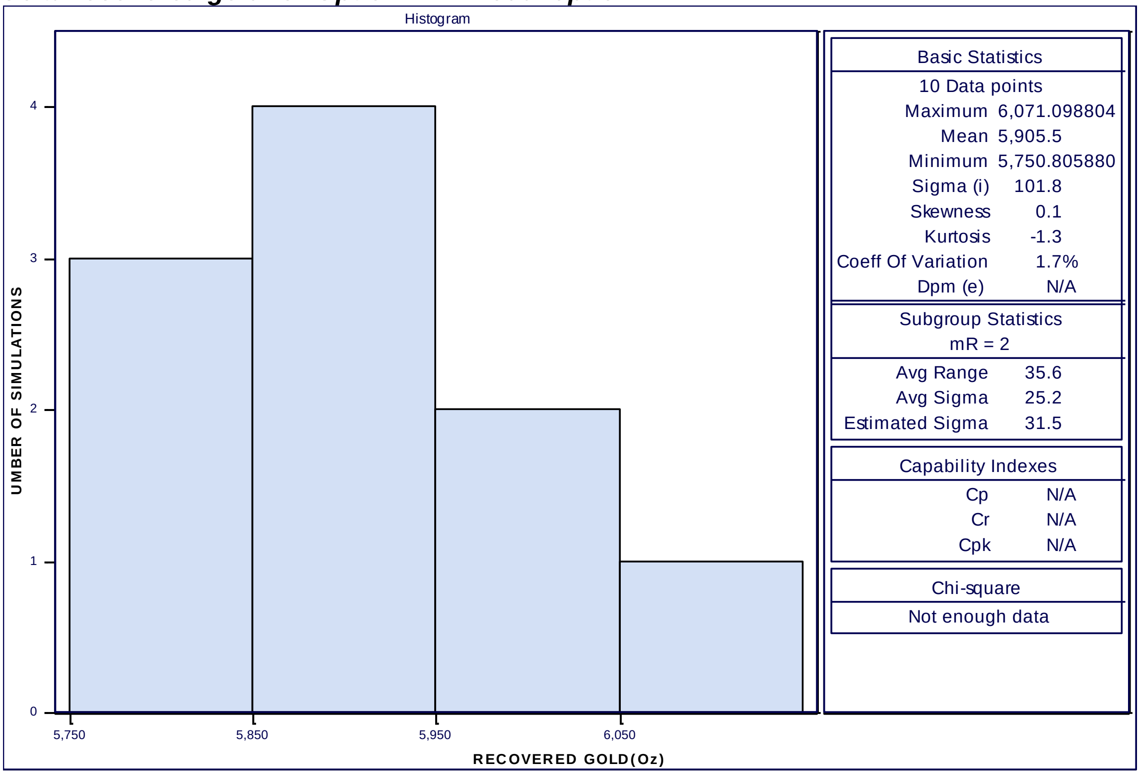
Geita recovered gold for Option 2



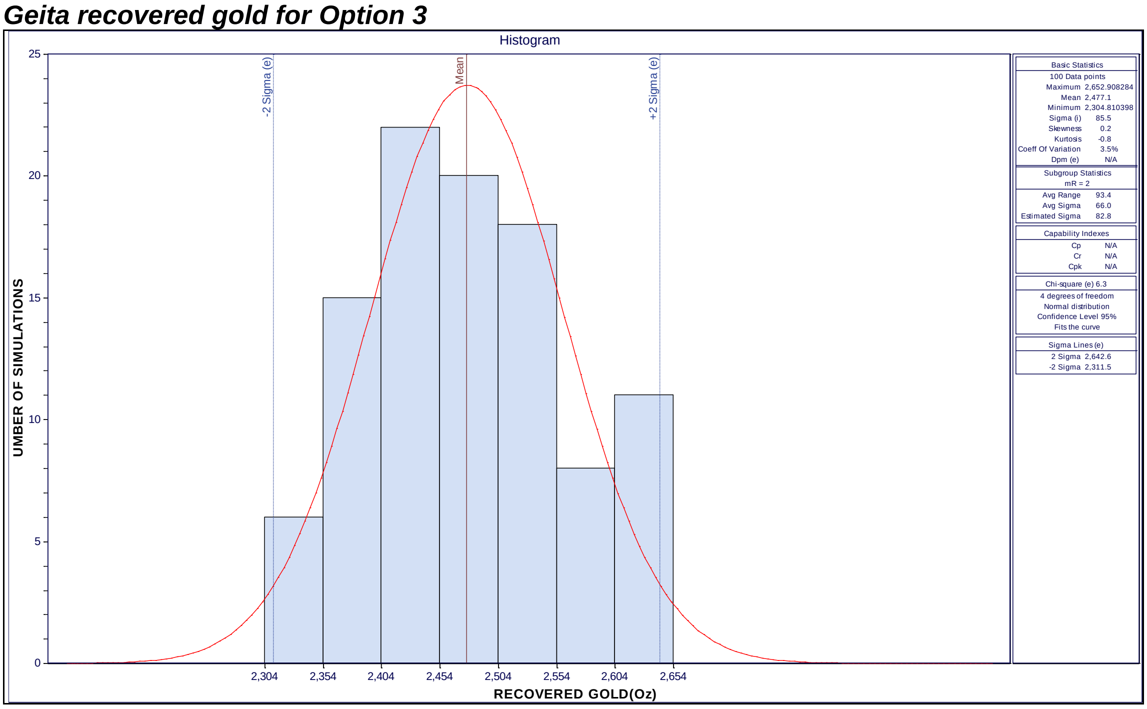
Geita recovered gold for Option 2 Bi-modal option 1



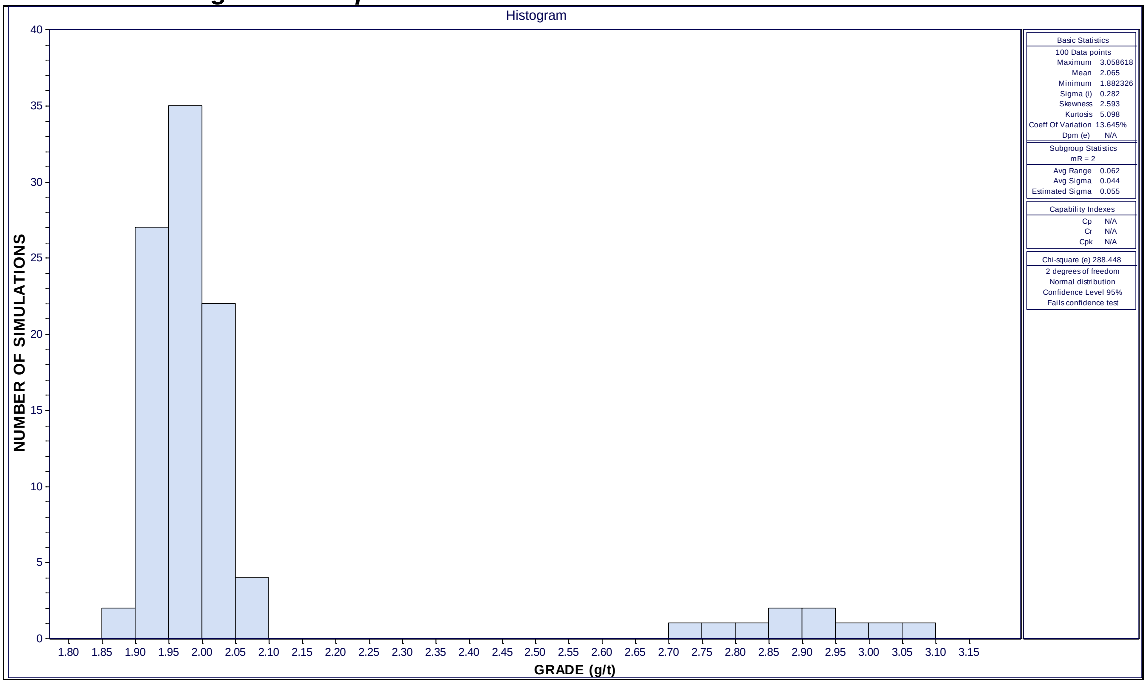
Geita recovered gold for Option 2 Bi-modal option 2



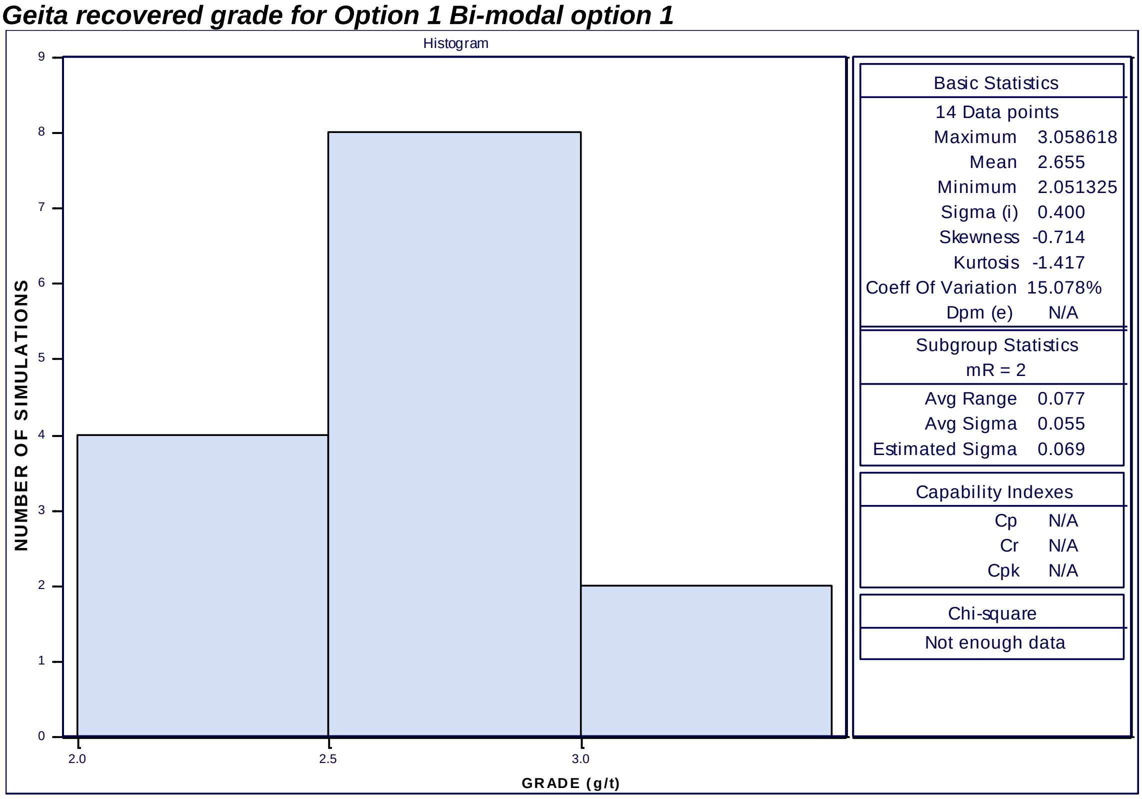
Geita recovered gold for Option 3



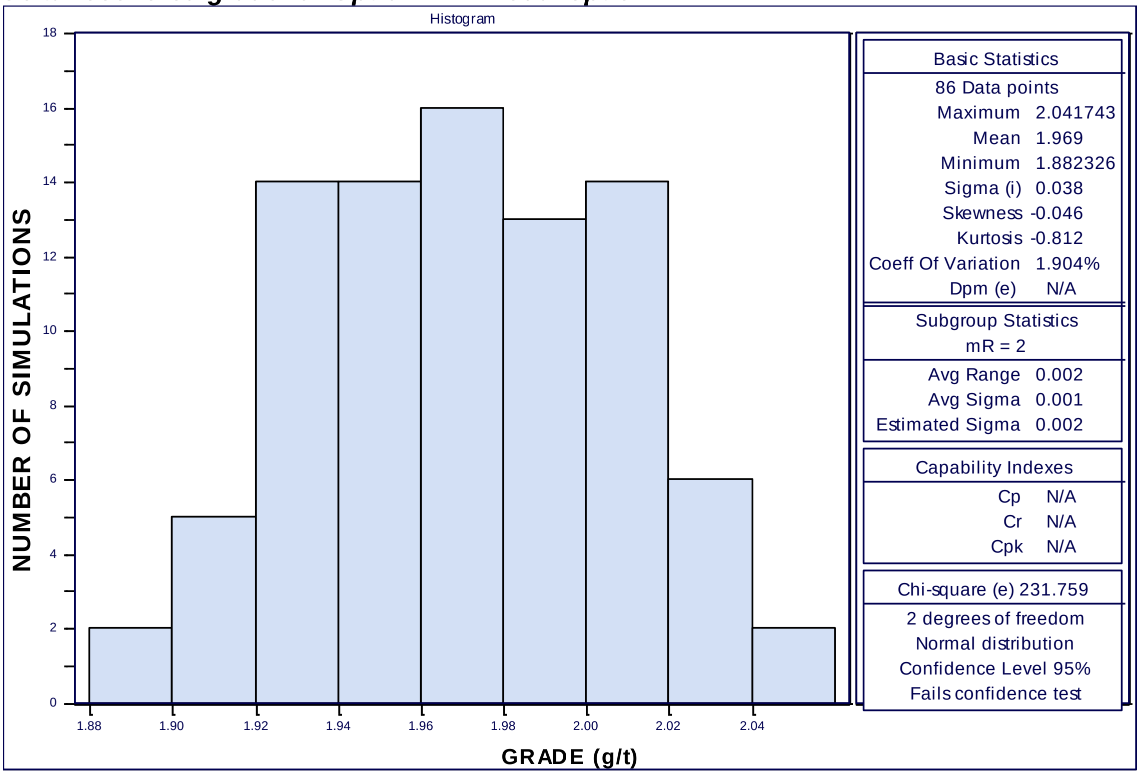
Geita recovered grade for Option 1



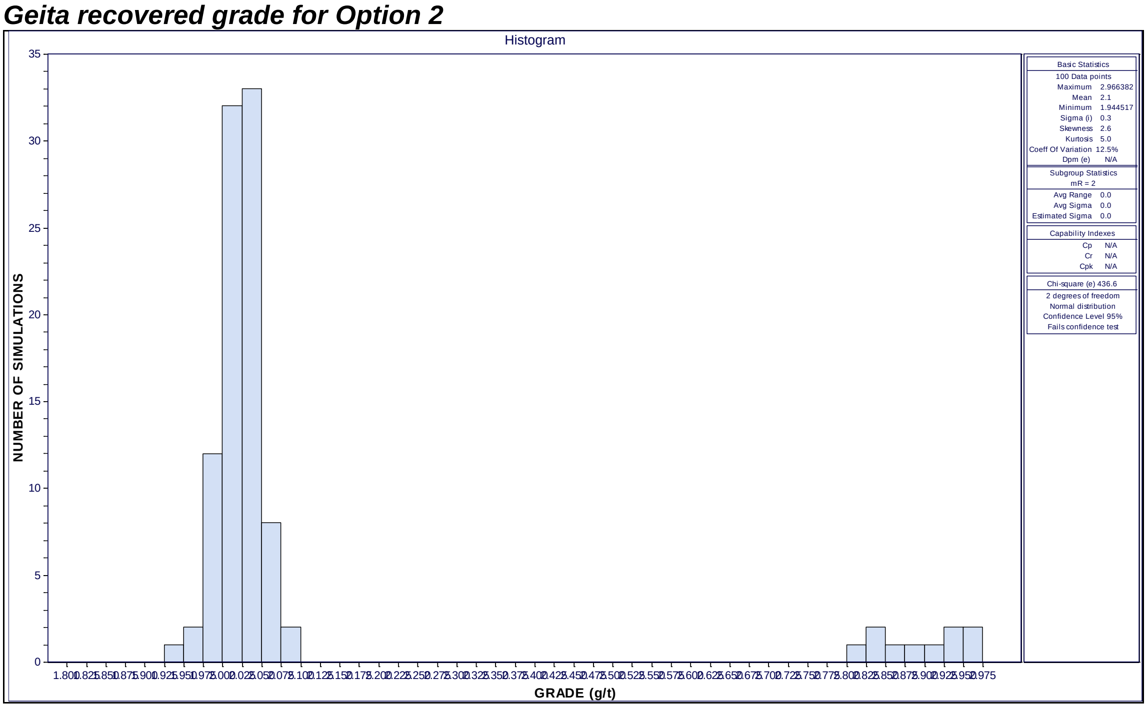
Geita recovered grade for Option 1 Bi-modal option 1



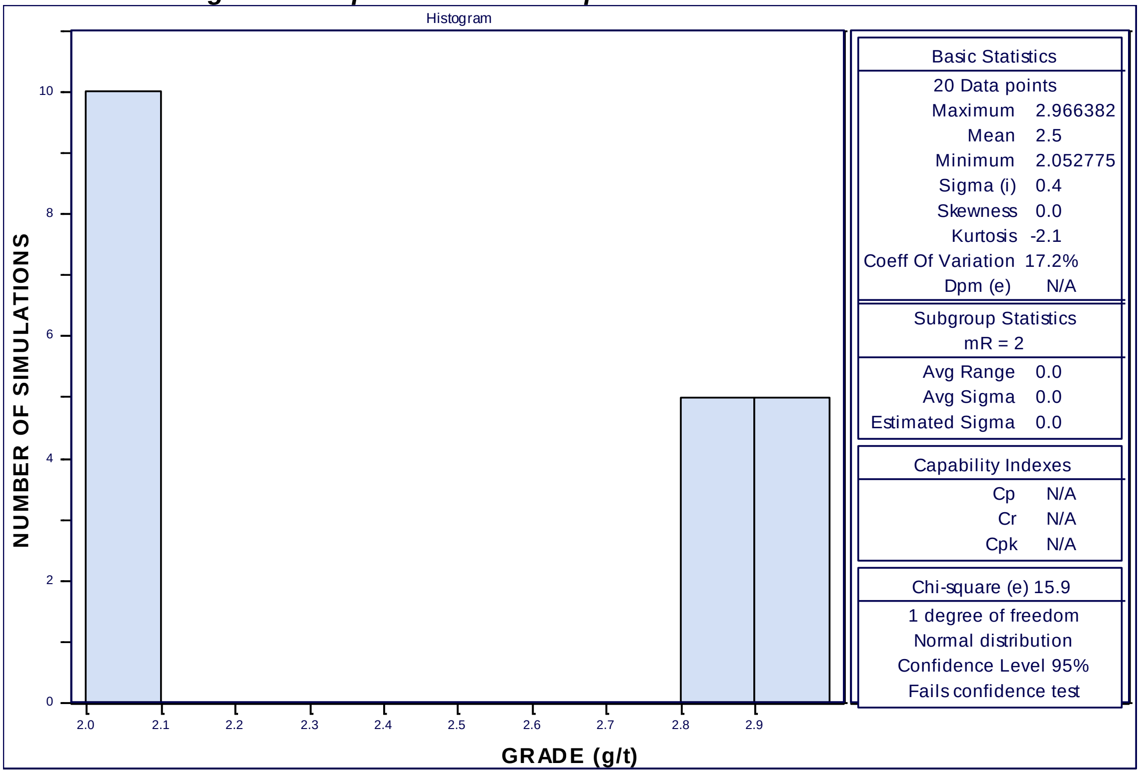
Geita recovered grade for Option 1 Bi-modal option 2



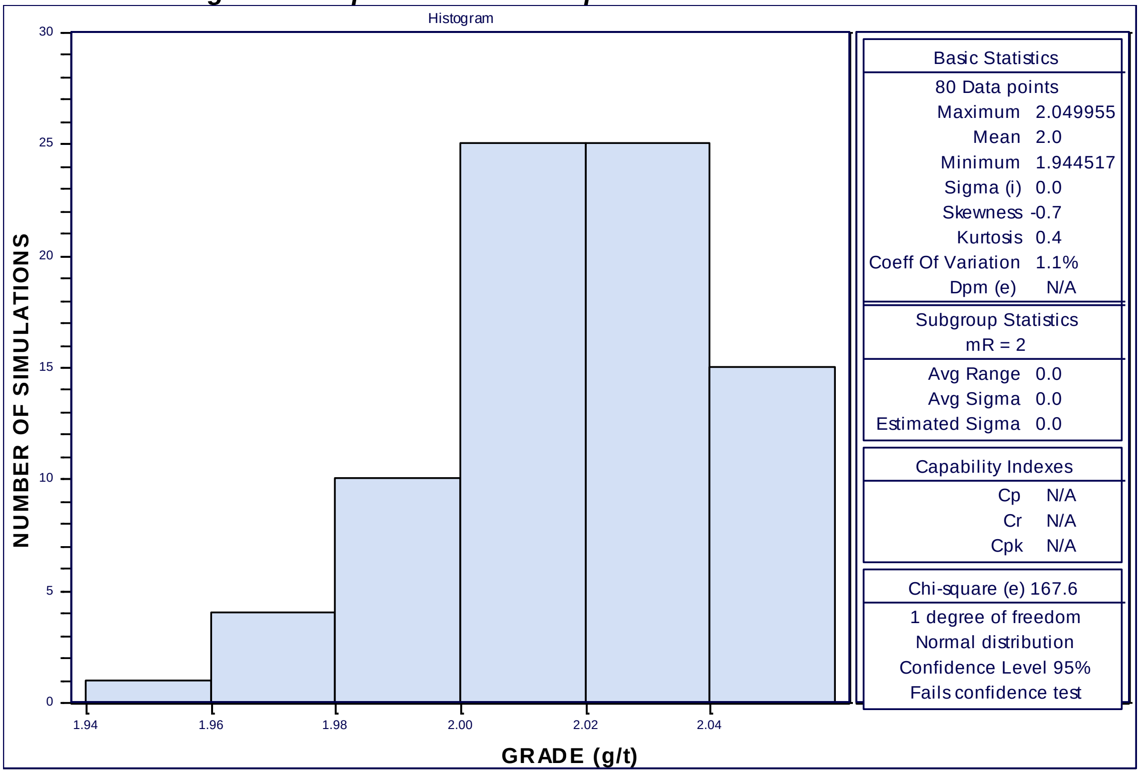
Geita recovered grade for Option 2



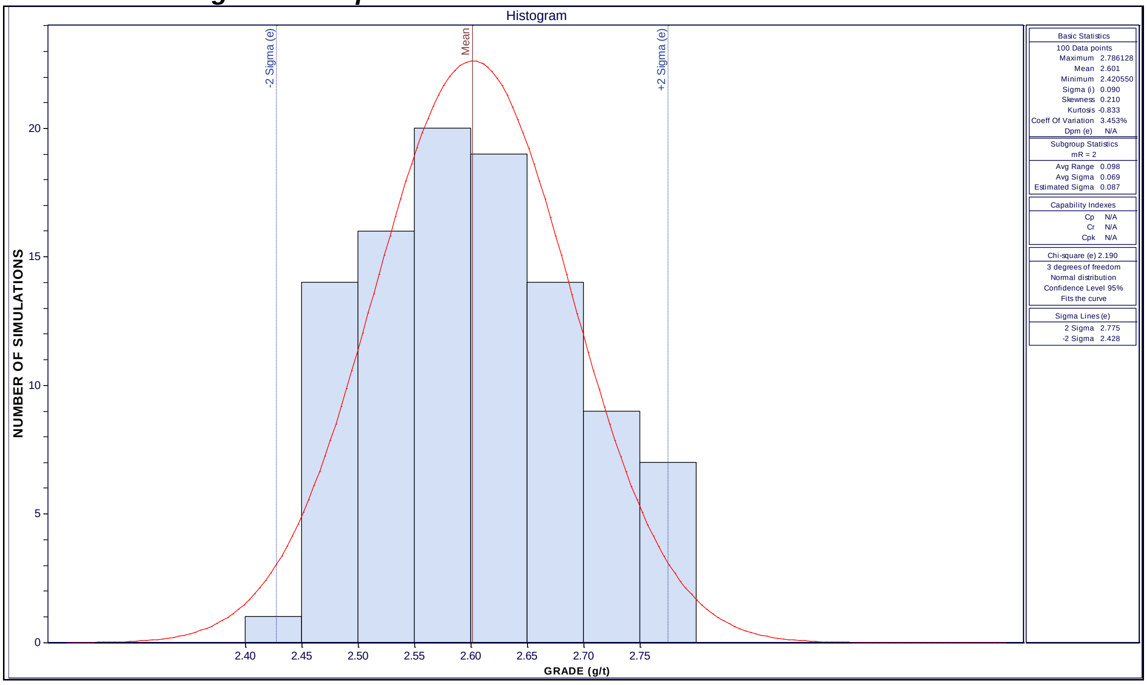
Geita recovered grade for Option 2 Bi-modal option 1



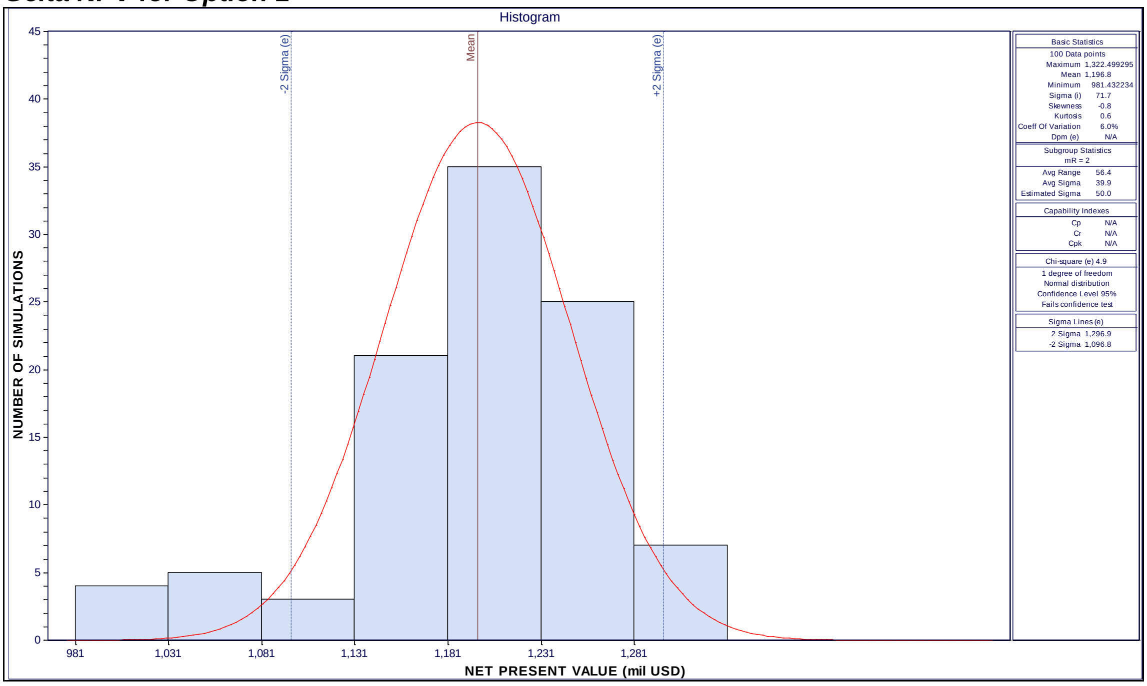
Geita recovered grade for Option 2 Bi-modal option 2



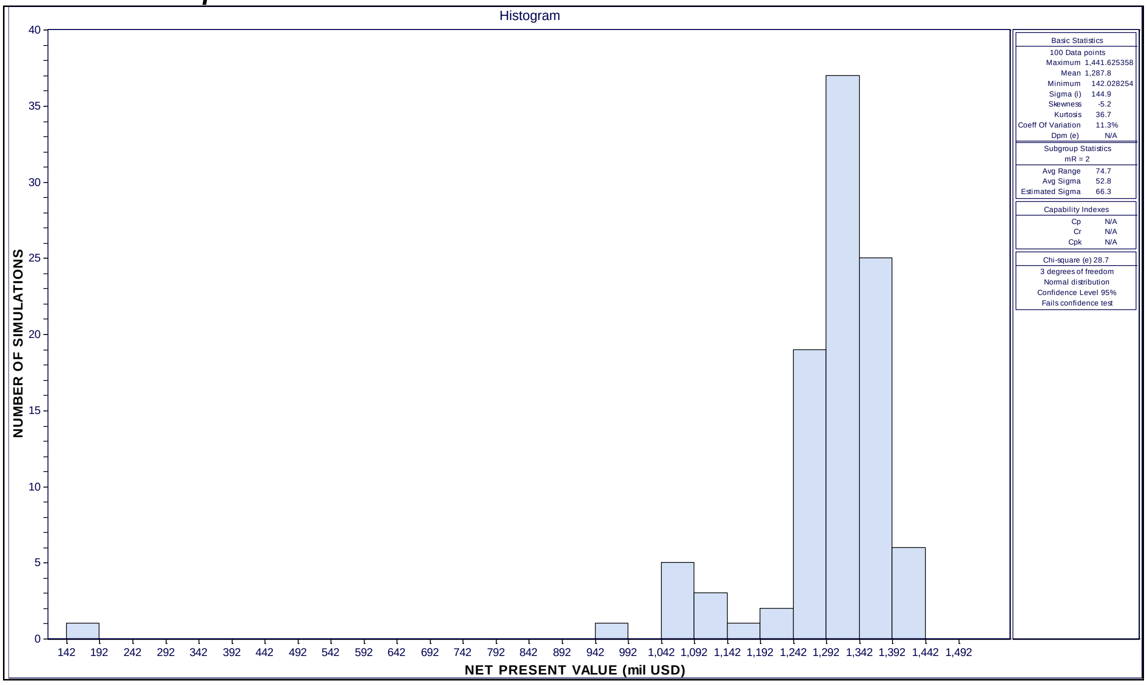
Geita recovered grade for Option 3



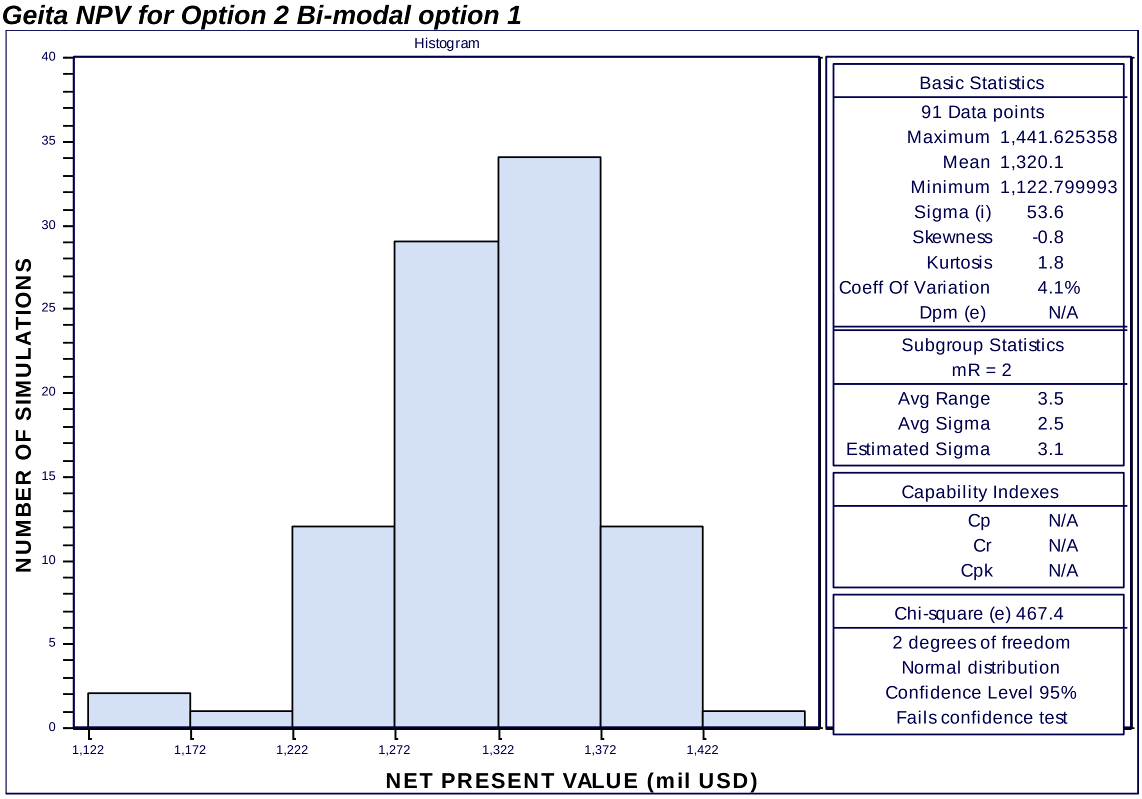
Geita NPV for Option 1



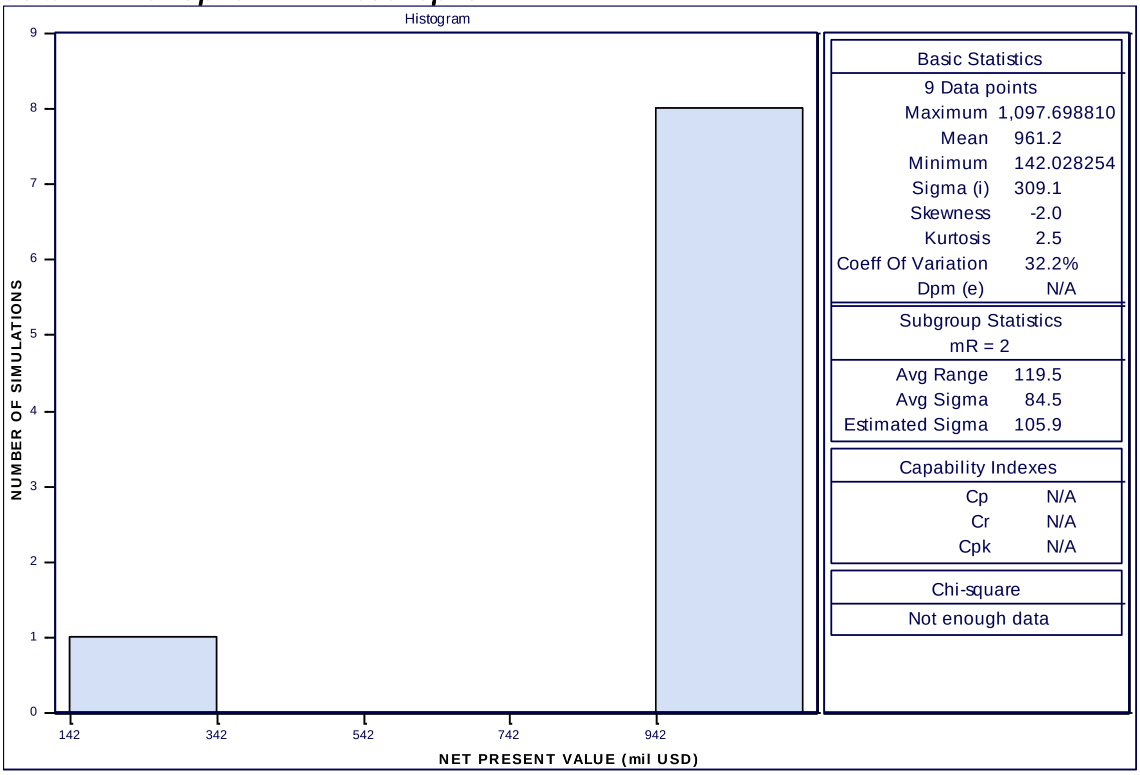
Geita NPV for Option 2



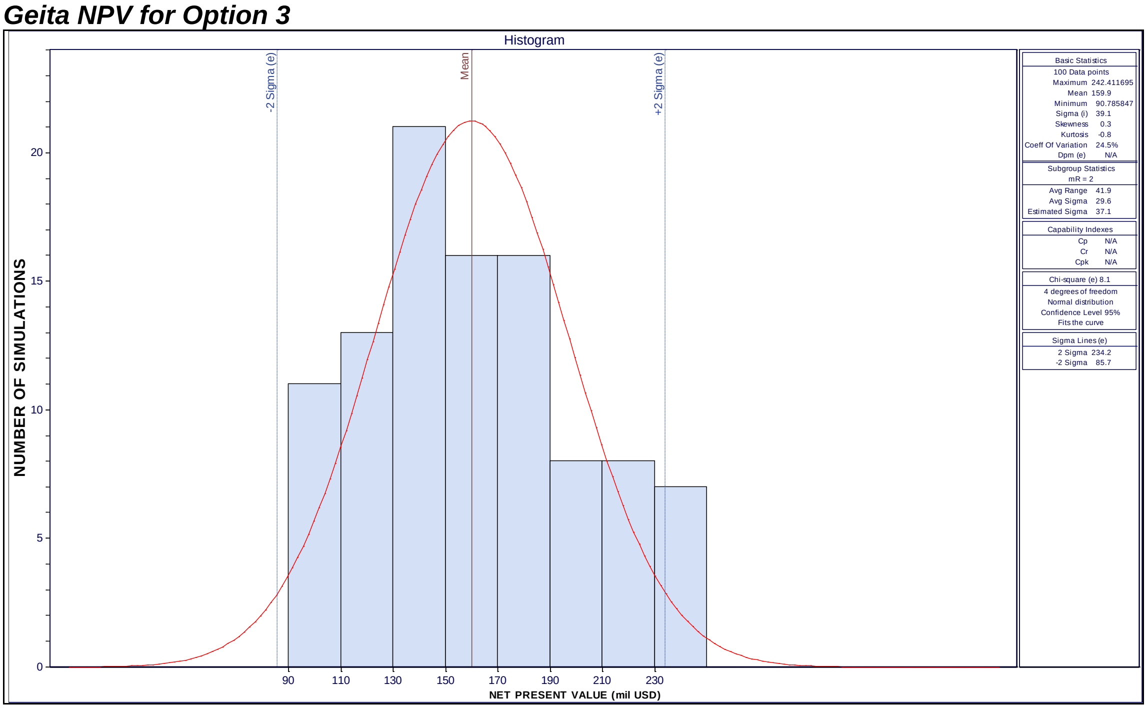
Geita NPV for Option 2 Bi-modal option 1



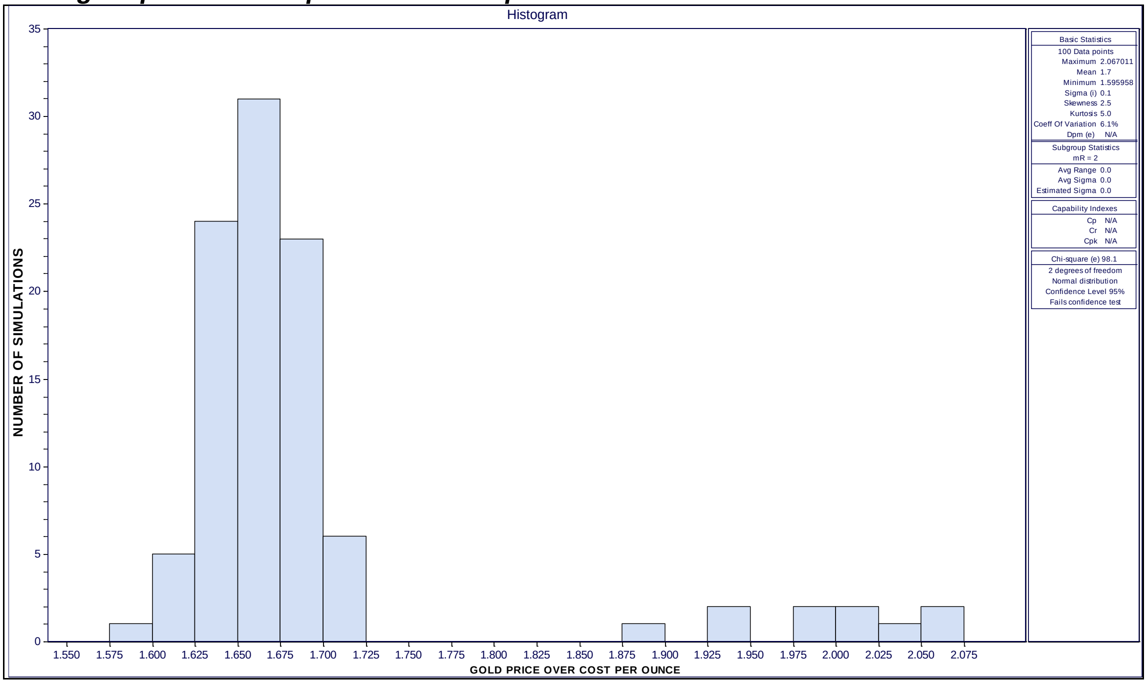
Geita NPV for Option 2 Bi-modal option 2



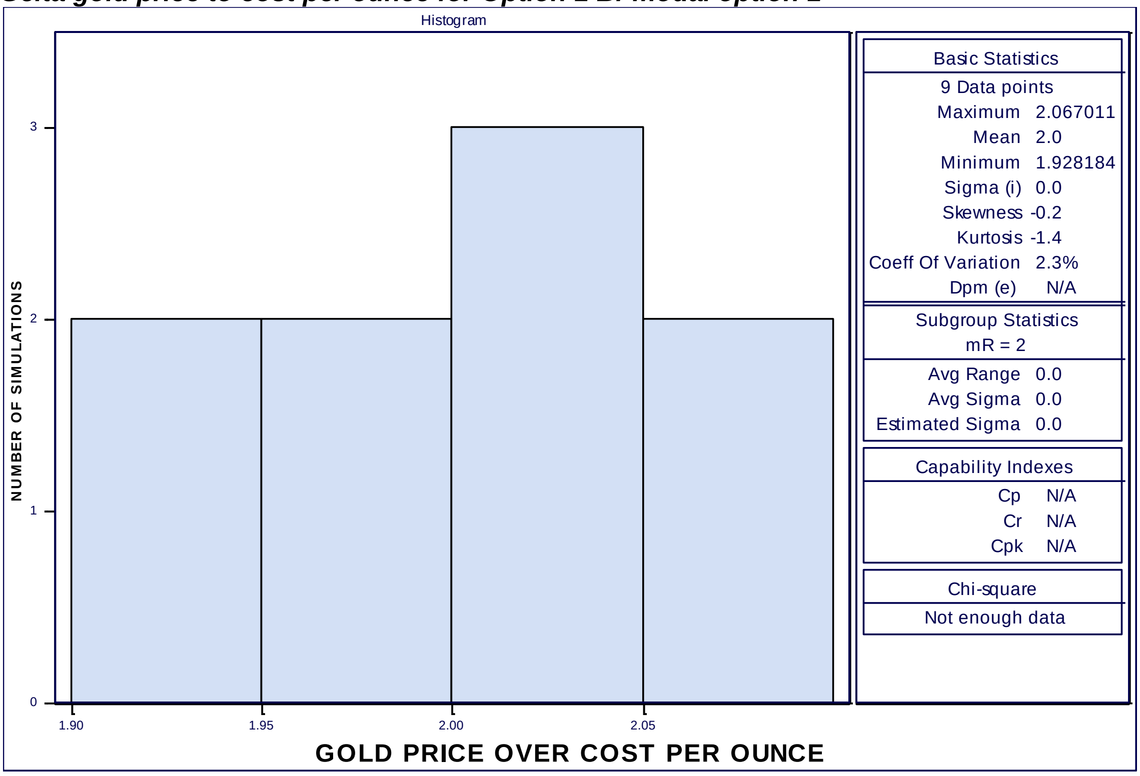
Geita NPV for Option 3



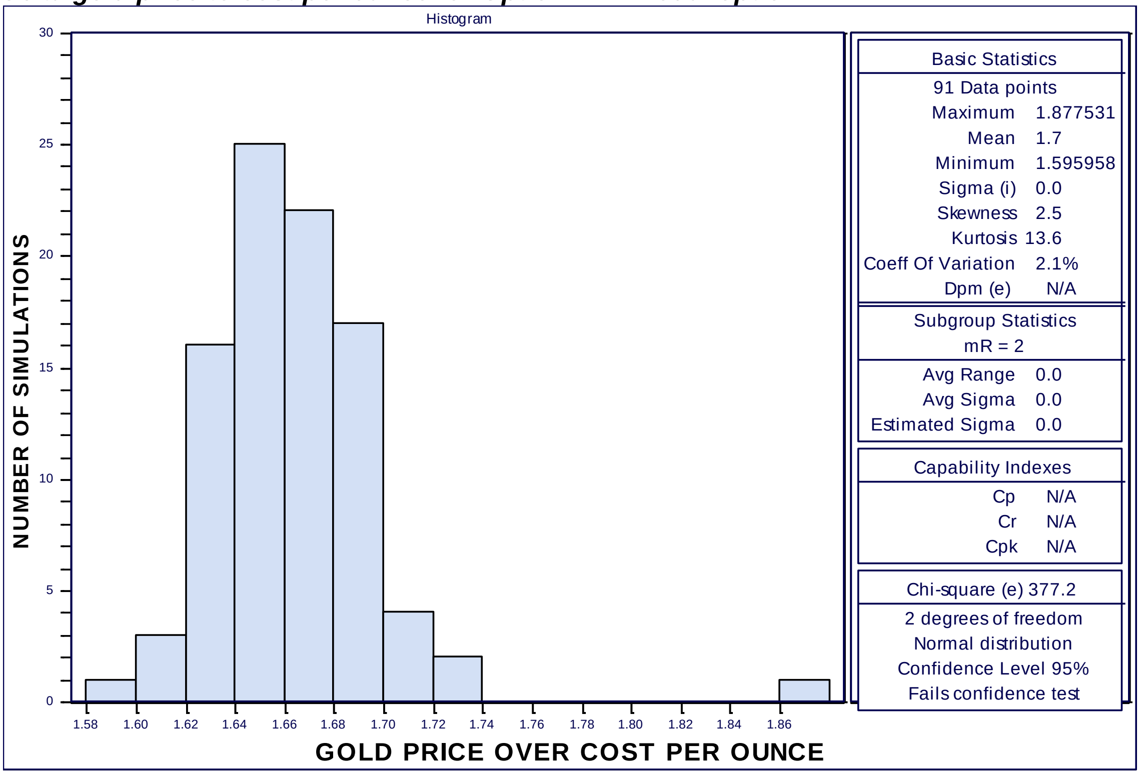
Geita gold price to cost per ounce for Option 1



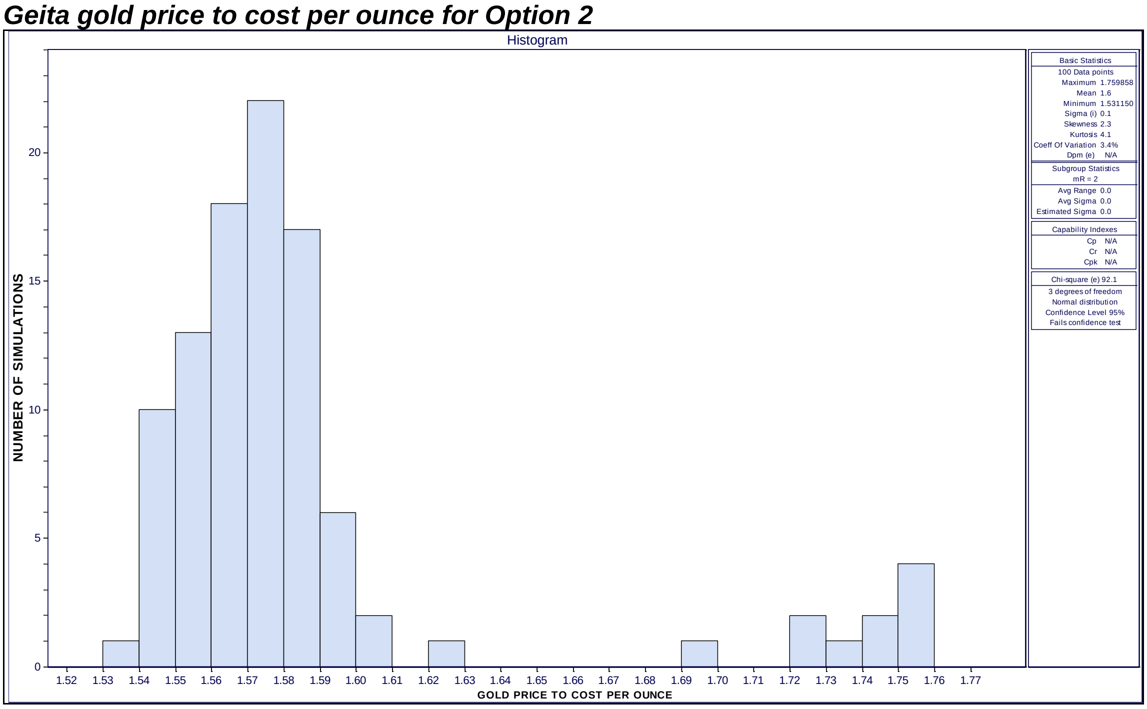
Geita gold price to cost per ounce for Option 1 Bi-modal option 1



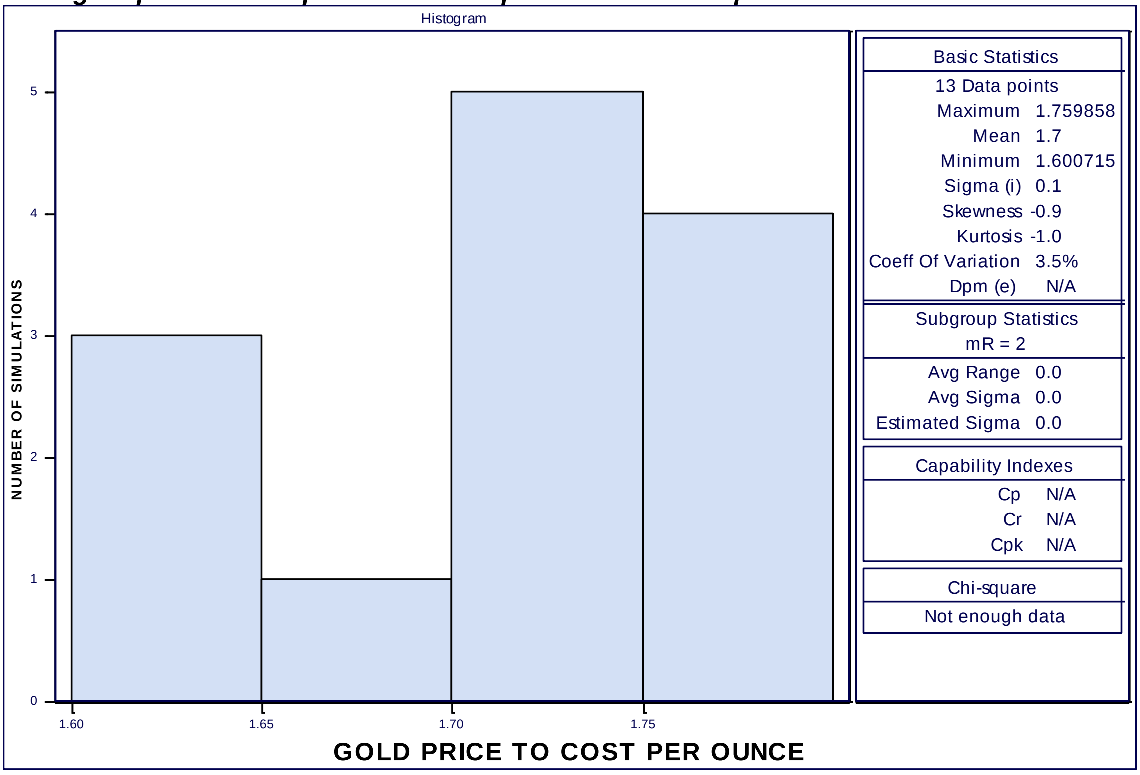
Geita gold price to cost per ounce for Option 1 Bi-modal option 2



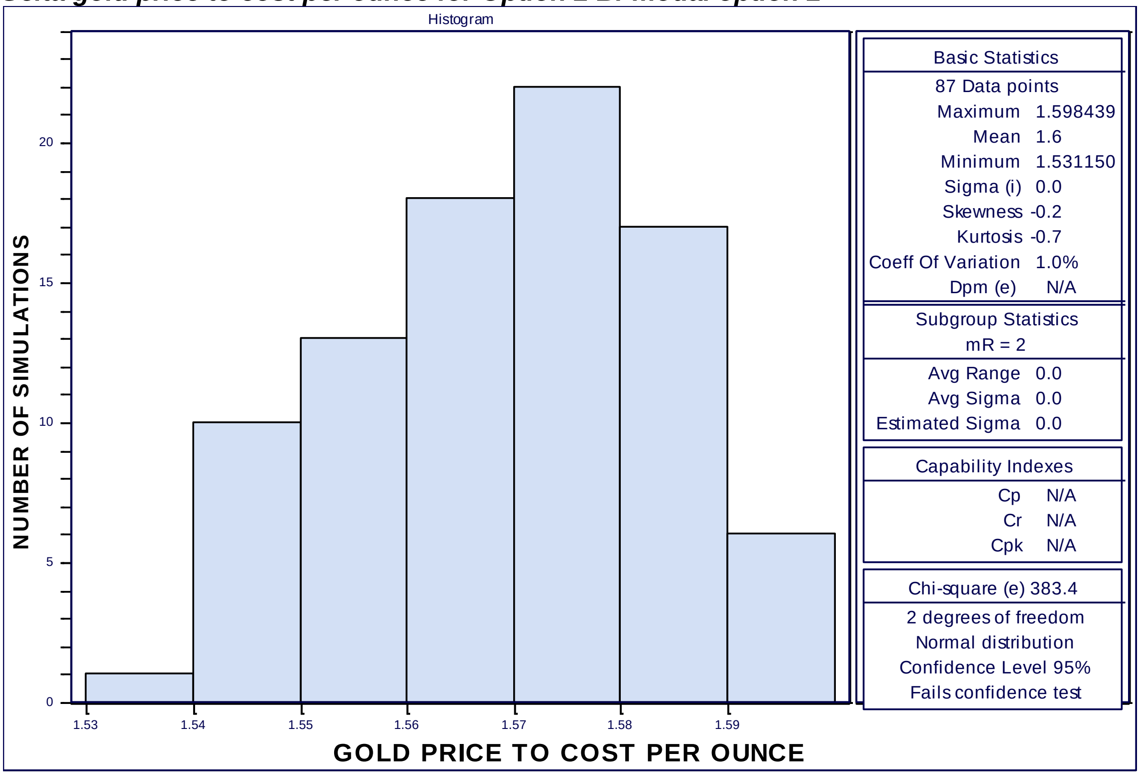
Geita gold price to cost per ounce for Option 2



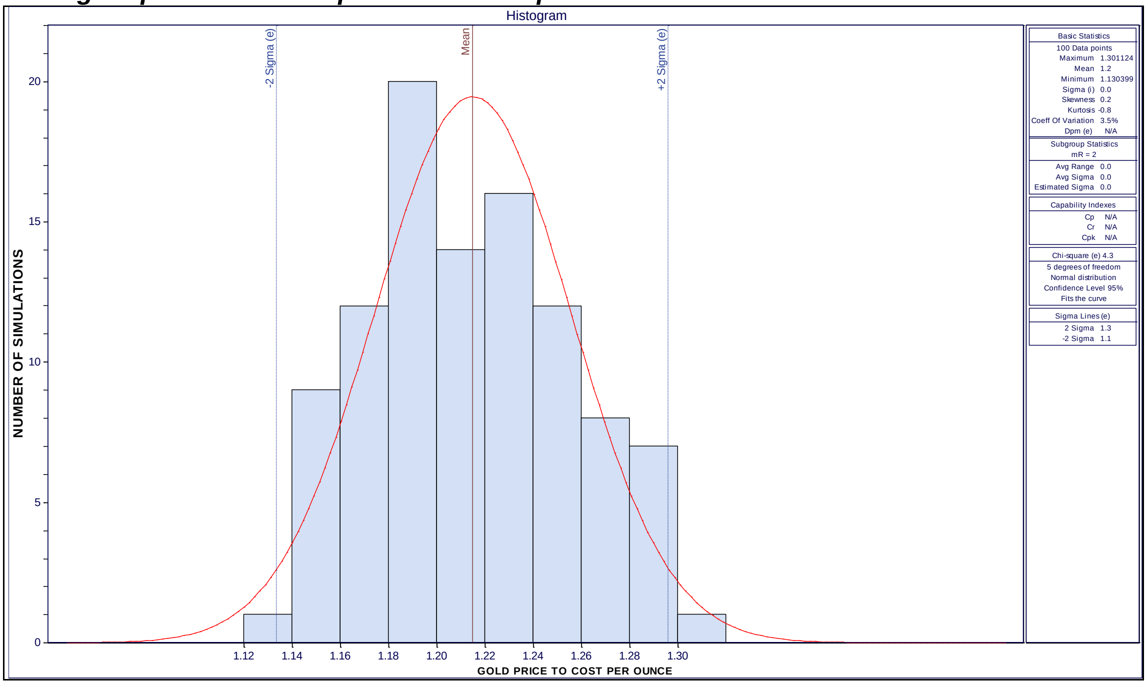
Geita gold price to cost per ounce for Option 2 Bi-modal option 1



Geita gold price to cost per ounce for Option 2 Bi-modal option 2

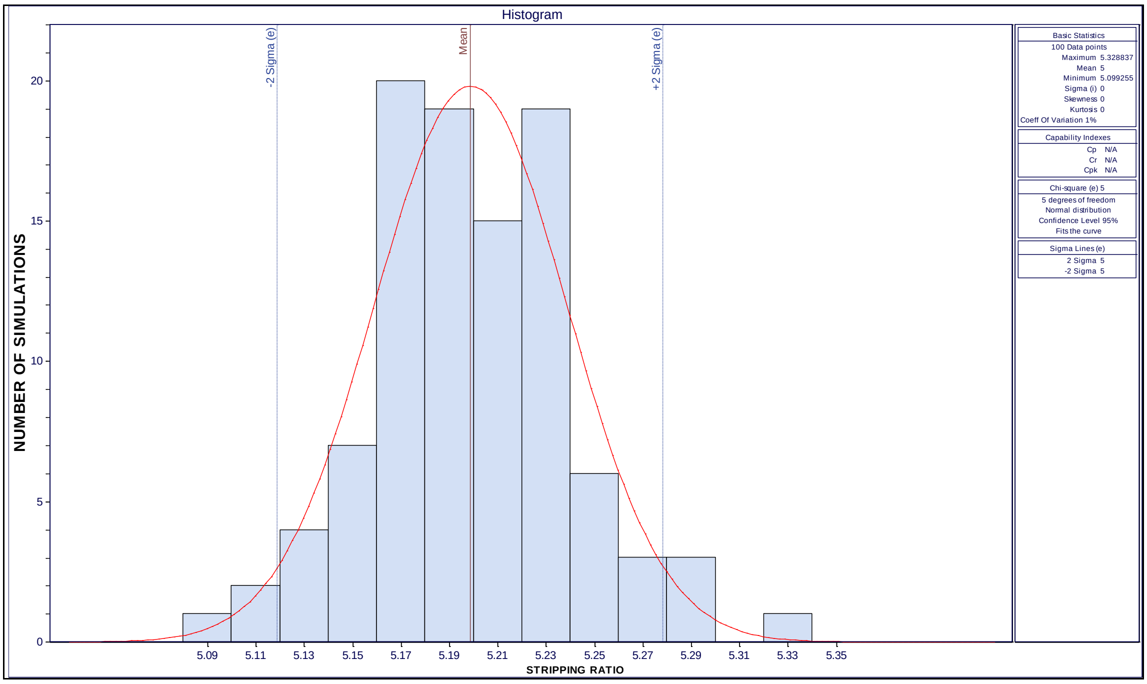


Geita gold price over cost per ounce for Option 3

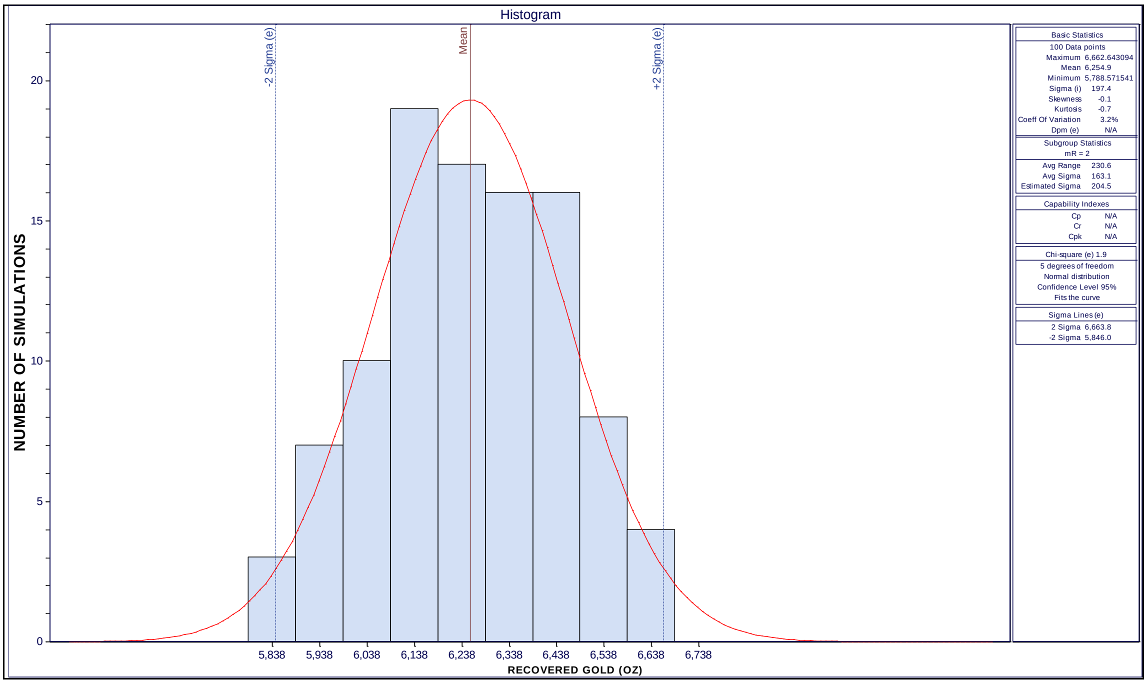


Appendix 12: Statistical summary for Morila transition indicators

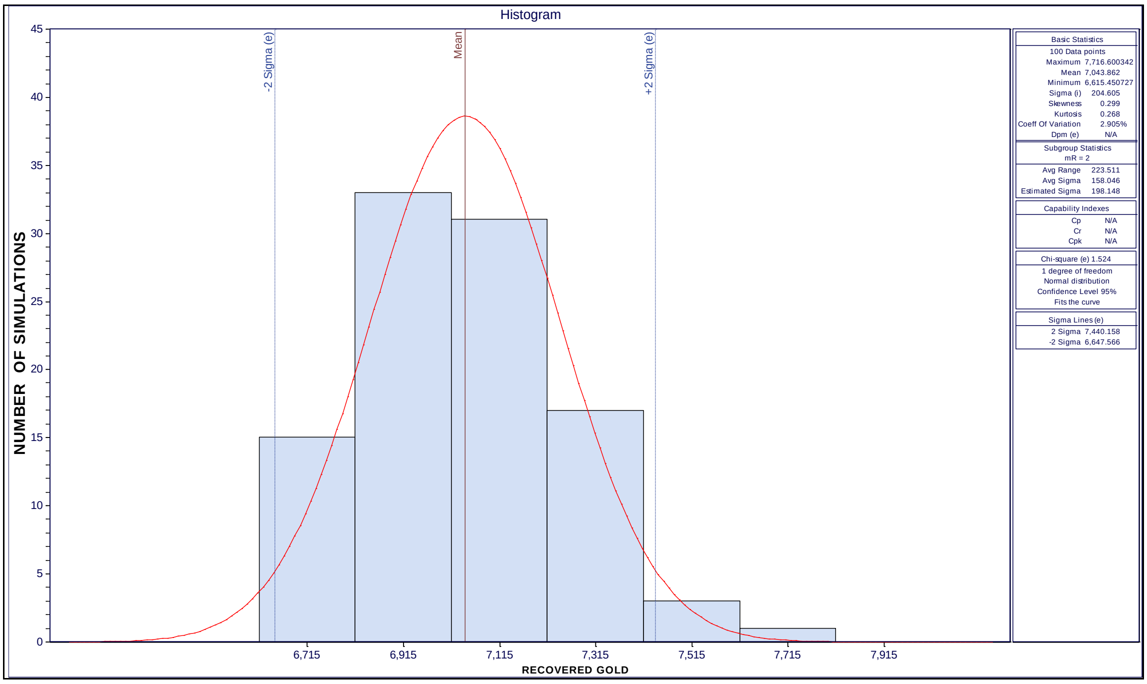
Morila stripping ratio for Option 1



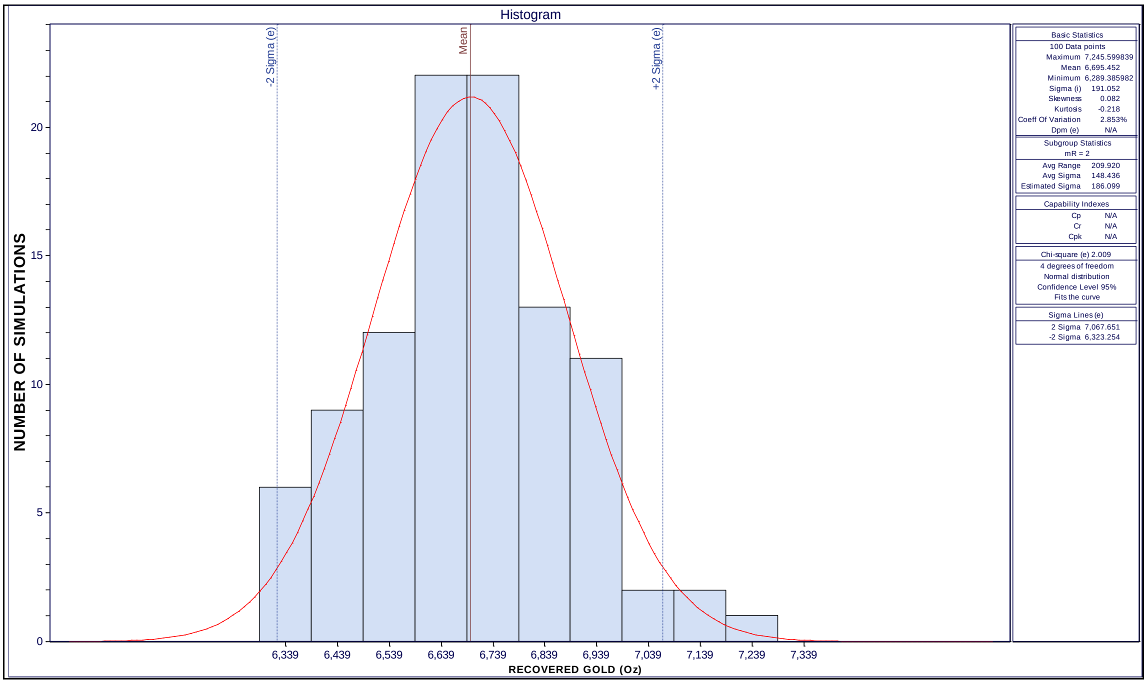
Morila recovered gold for Option 1



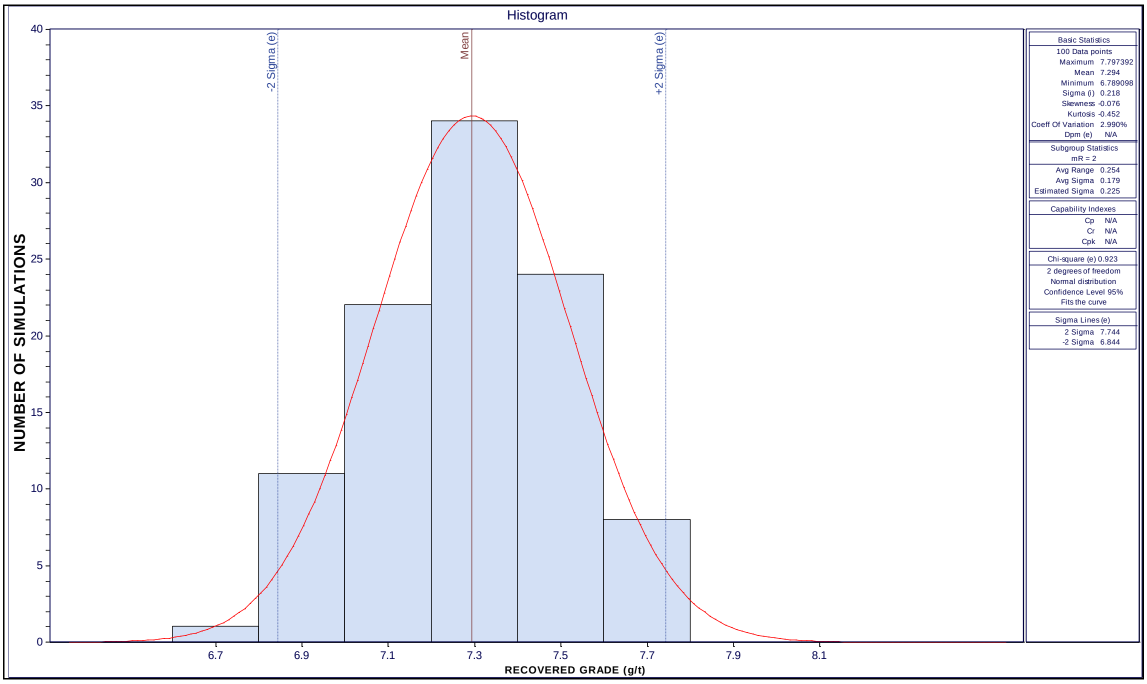
Morila recovered gold for Option 2



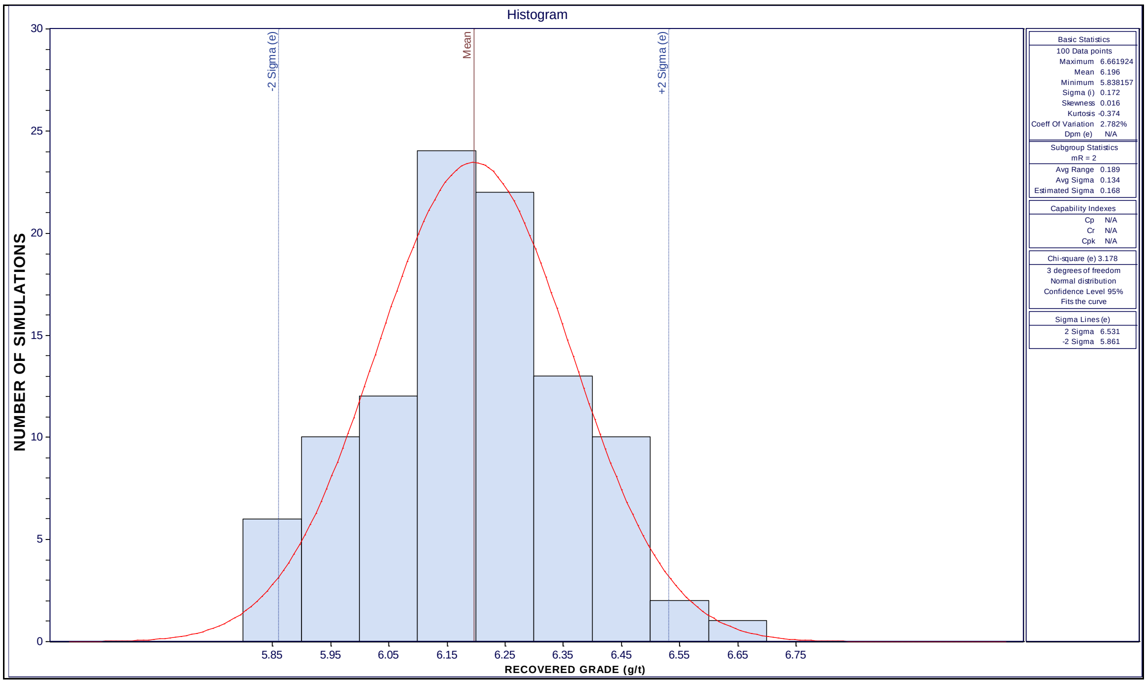
Morila recovered gold for Option 3



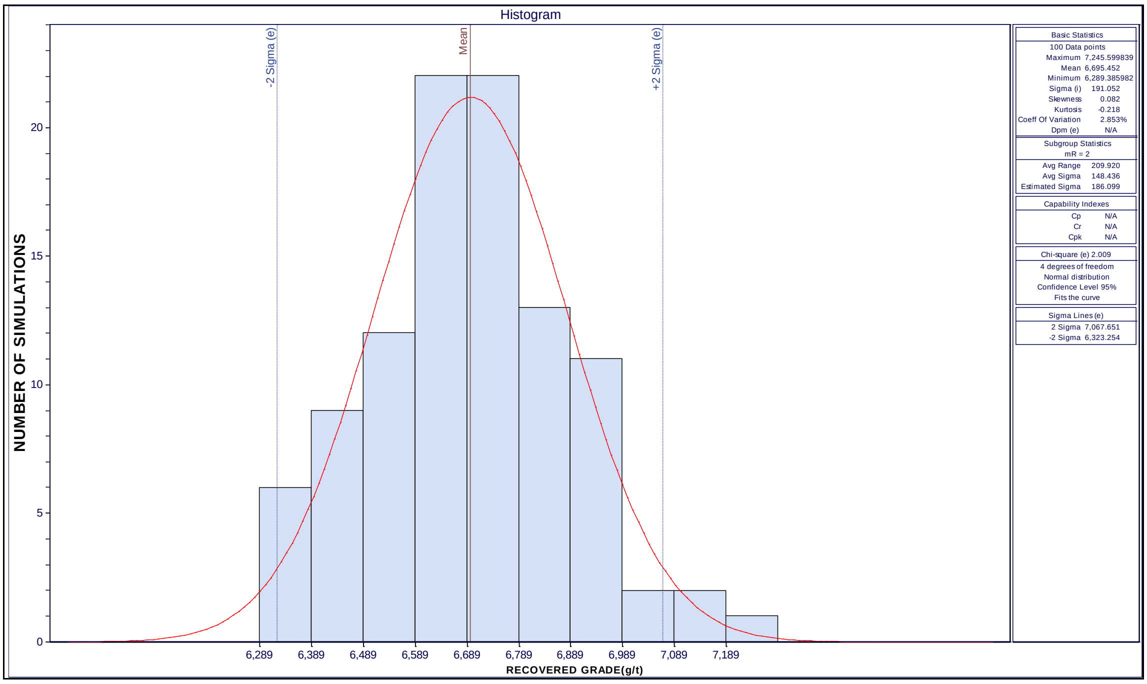
Morila recovered grade for Option 1



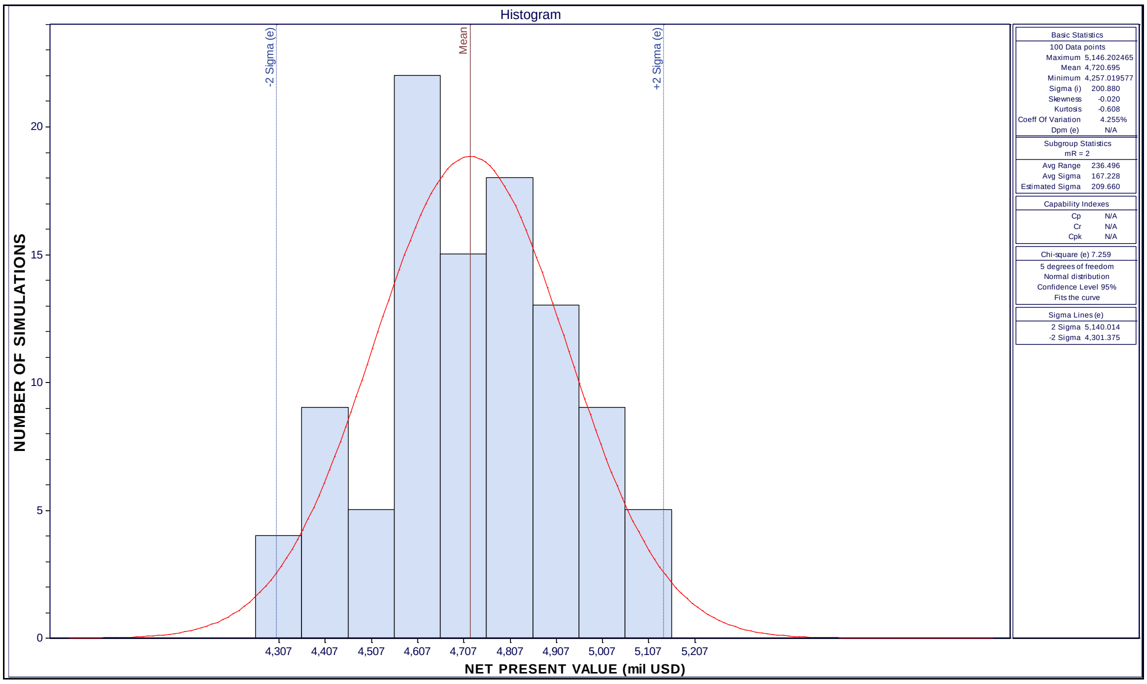
Morila recovered grade for Option 2



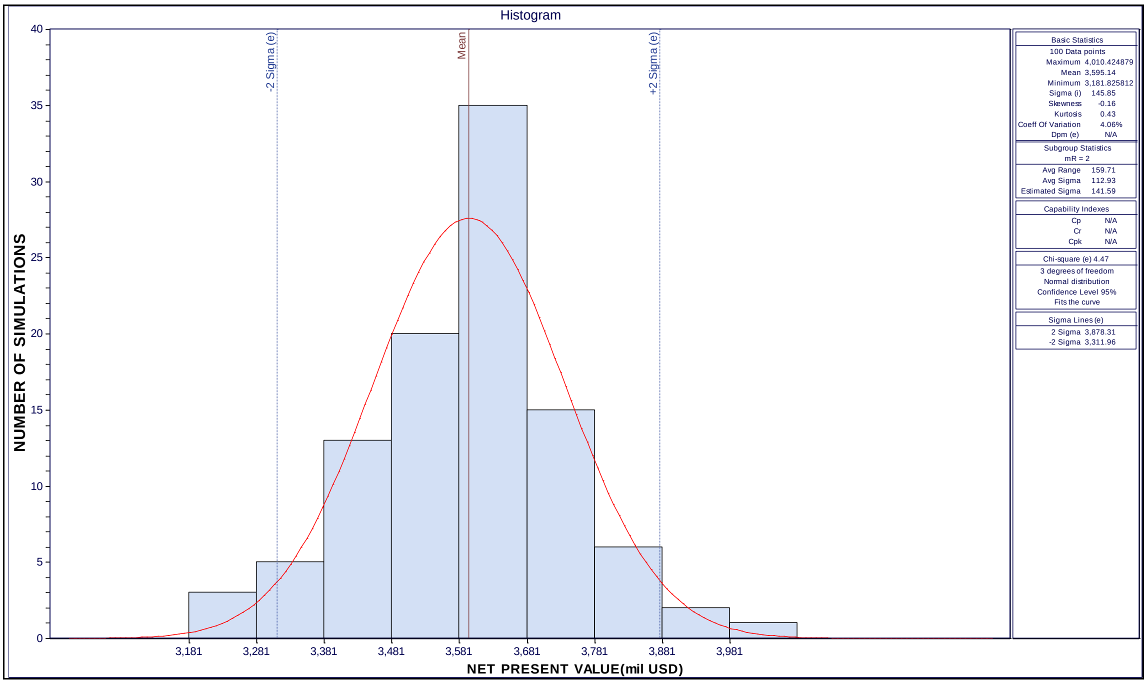
Morila recovered grade for Option 3



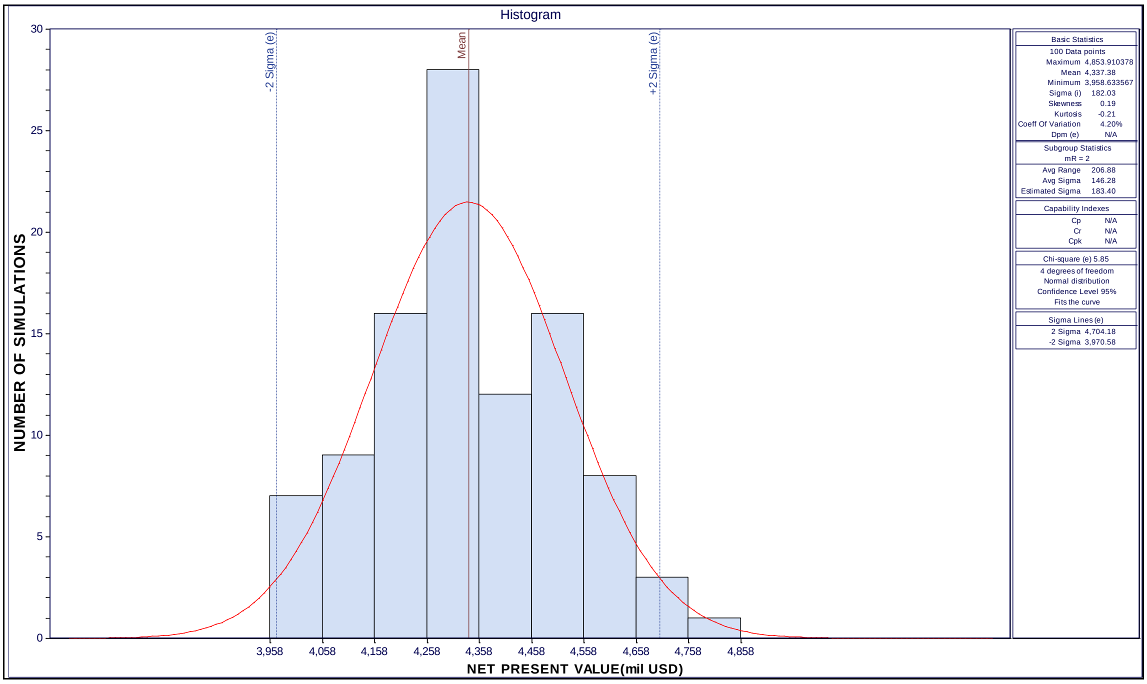
Morila NPV for Option 1



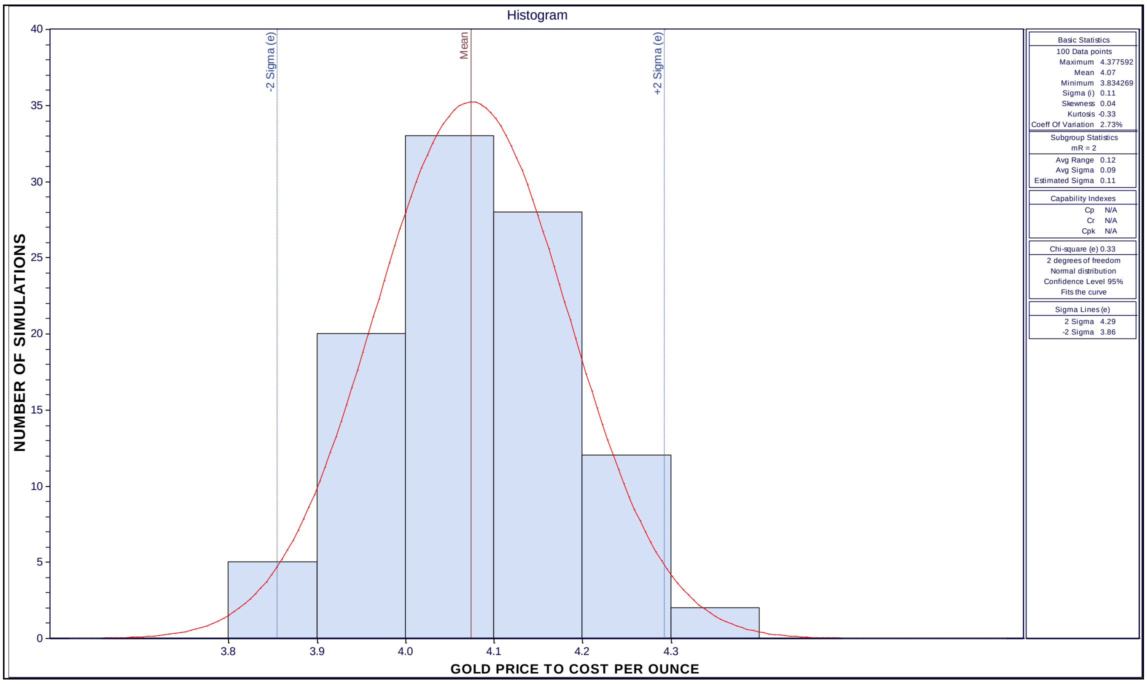
Morila NPV for Option 2



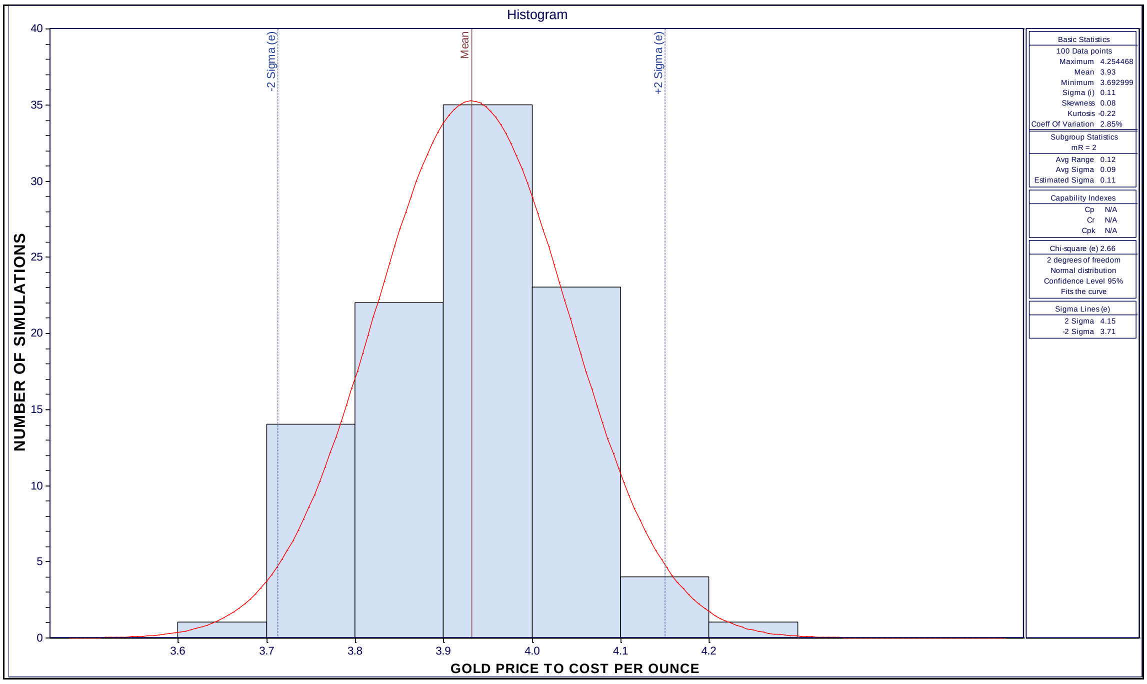
Morila NPV for Option 3



Morila gold price over cost per ounce for Option 1



Morila gold price over cost per ounce for Option 2



Morila gold price over cost per ounce for Option 3