

**RUSORO MINING LTD.
NI 43-101 TECHNICAL REPORT
FEASIBILITY STUDY
EXPANSION OF GOLD PRODUCTION AT
CHOCO 10 AND INCREIBLE 6

BOLIVAR STATE, VENEZUELA**

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Table of Contents

	Page
1.0 SUMMARY	1
1.1 INTRODUCTION	2
1.2 PROPERTY DESCRIPTION	2
1.3 HISTORY	3
1.4 GEOLOGY AND MINERALIZATION	4
1.5 EXPLORATION AND OTHER PROGRAMS	4
1.6 MINERAL RESOURCE ESTIMATE	5
1.7 MINERAL RESERVE ESTIMATE	6
1.8 MINING METHODS	7
1.9 METALLURGICAL TESTWORK	7
1.10 PROCESSING	8
1.11 PROJECT INFRASTRUCTURE	9
1.12 ENVIRONMENTAL AND SOCIAL ASPECTS	9
1.13 PROJECT IMPLEMENTATION	9
1.14 CAPITAL COSTS	11
1.15 OPERATING COSTS	11
1.16 ECONOMIC ANALYSIS	11
1.16.1 Basis of Evaluation	11
1.16.2 Macro-Economic Assumptions	12
1.16.3 Mine Production Schedule	13
1.16.4 Base Case Evaluation	16
1.16.5 Sensitivity Study	17
1.17 CONCLUSIONS	18
1.18 RECOMMENDATIONS	18
1.18.1 Geology	18
1.18.2 Geotechnical	19
1.18.3 Mining - Waste Storage	20
1.18.4 Metallurgy	20
1.18.5 Schedule	20
1.18.6 Power Supply	20
1.18.7 Cost Estimates	20
2.0 INTRODUCTION	22
2.1 TERMS OF REFERENCE	22
2.1.1 Qualified Persons	23
2.1.2 Site Visits	23
2.1.3 Study Preparation	23
2.2 UNITS AND CURRENCY	24
3.0 RELIANCE ON OTHER EXPERTS	26
4.0 PROPERTY DESCRIPTION AND LOCATION	27
4.1 LOCATION	27
4.2 CLAIMS	28

4.2.1	Choco Concession.....	28
4.2.2	Increible 6 Concession	29
4.3	ROYALTIES	29
4.4	PERMITS AND LICENSES	29
5.0	ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY	30
5.1	ACCESS	30
5.2	CLIMATE AND PHYSIOGRAPHY	30
5.3	INFRASTRUCTURE	30
6.0	HISTORY	32
6.1	CHOCO 10	32
6.2	INCREIBLE 6	32
7.0	GEOLOGICAL SETTING AND MINERALIZATION.....	34
7.1	REGIONAL GEOLOGY	34
7.2	LOCAL GEOLOGY	36
7.3	LOCAL DEPOSITS AND MINERAL OCCURRENCES	38
7.3.1	Choco 10 Deposits Geology	38
7.3.2	Rosika, Coacia and Pisolita.....	40
7.3.3	Villa Balazo-Karolina	40
7.3.4	Increible 6	41
7.4	MINERALIZATION.....	41
7.4.1	Choco 10	41
7.4.2	Increible 6	42
7.5	ALTERATION	43
8.0	DEPOSIT TYPES	44
9.0	EXPLORATION	45
9.1	CHOCO 10	45
9.2	INCREIBLE 6	46
10.0	DRILLING	47
10.1	CHOCO 10	47
10.1.1	Core Statistics	49
10.2	INCREIBLE 6	49
10.3	CORE LOGGING AND SAMPLE PROCEDURES.....	51
10.3.1	Choco 10	51
10.3.2	Increible 6	53
10.4	SURVEYING OF DRILL HOLES.....	54
11.0	SAMPLE PREPARATION, ANALYSES AND SECURITY	55
11.1	SAMPLE PREPARATION AND QA/QC BEFORE DISPATCH OF SAMPLES	55
11.1.1	Sample Preparation, Logistics and QA/QC	56
11.2	LABORATORY SAMPLE PREPARATION, AND ANALYSES	57
11.2.1	Density Analysis	58

11.2.2	Security	59
12.0	DATA VERIFICATION	61
12.1	SUMMARY	61
12.2	DATA VALIDATION AND DATABASE	61
12.3	MICON DATA VERIFICATION.....	66
13.0	MINERAL PROCESSING AND METALLURGICAL TESTWORK	67
13.1	PREVIOUS METALLURGICAL TESTWORK AND FLOWSHEET DEVELOPMENT.....	67
13.1.1	McClelland Testwork.....	68
13.1.2	Amdel Testwork.....	68
13.1.3	Lakefield Testwork	69
13.2	PREVIOUS TESTWORK RESULTS	69
13.2.1	Grinding	69
13.2.2	Gravity Concentration.....	69
13.2.3	Cyanide Leaching.....	70
13.2.4	Discussion of Results	72
13.3	TEST WORK BY INSPECTORATE AMERICA CORP. (PRA), 2008 - 2010	73
13.3.1	Sample Information & Test Procedures	73
13.3.2	Cyanide Leaching.....	74
13.3.3	Gravity Concentration.....	74
13.3.4	Flotation	75
13.3.5	Bond Ball Mill Work & Abrasion Index Determinations.....	75
13.3.6	Results and Discussion.....	75
13.3.7	Conclusions and Recommendations	84
13.4	AMMTEC (AUSTRALIA) TESTING.....	84
13.4.1	Comminution and Confirmatory Leach Testwork	84
13.5	TESTWORK CONCLUSIONS	88
14.0	MINERAL RESOURCE ESTIMATES.....	90
14.1	CHOCO 10	90
14.1.1	Previous Estimate.....	90
14.1.2	Methodology	91
14.1.3	Block Model.....	96
14.1.4	Interpolation Method.....	96
14.1.5	Block Model Report.....	97
14.1.6	Block Model Validation.....	97
14.1.7	Mineral Resources.....	98
14.1.8	Choco 10 Discussion.....	104
14.2	INCREDIBLE 6.....	105
14.2.1	Previous Estimate.....	105
14.2.2	Methodology	106
14.2.3	Block Models	111
14.2.4	Interpolation Method.....	112
14.2.5	Block Model Report.....	112

14.2.6	Block Model Validation	113
14.2.7	Increible 6 Mineral Resource Estimate	114
15.0	MINERAL RESERVE ESTIMATES	119
16.0	MINING METHODS	120
16.1	MINING OPERATIONS	120
16.2	MINE DESIGN	120
16.2.1	Open Pit Optimization	120
16.3	LIFE OF MINE PIT DESIGN	126
16.4	MINE PRODUCTION SCHEDULE	126
16.5	MINING EQUIPMENT	129
16.6	WASTE ROCK STORAGE	130
16.7	ORE STOCKPILES	131
17.0	RECOVERY METHODS	132
17.1	CURRENT PROCESSING FACILITIES	132
17.1.1	Crushing and Grinding	134
17.1.2	CIP Leaching and Gold Recovery	134
17.1.3	Sampling Analysis, Gold Accounting and Security	134
17.1.4	Production	135
17.2	PROPOSED EXPANSION OPTIONS CONSIDERED	135
17.3	PROCESS PLANT DESIGN BASIS AND DESCRIPTION	139
17.3.1	General	139
17.3.2	Process Plant Design Basis	139
17.3.3	Throughput and Availability	139
17.3.4	Processing Strategy	140
17.3.5	Unit Process Selection	143
18.0	PROJECT INFRASTRUCTURE	157
18.1	POWER SUPPLY	157
18.2	FUEL AND LUBE STORAGE	157
18.3	FIRE PROTECTION	157
18.4	WATER SUPPLY AND TREATMENT	158
18.4.1	Potable Water	158
18.4.2	Sewage Treatment	158
18.4.3	Site Drainage	158
18.5	IN-PLANT ROADS	158
18.6	COMMUNICATIONS	158
18.7	SECURITY	159
18.8	SITE BUILDINGS	159
18.8.1	Plant Administration Building	159
18.8.2	Main Administration Building	159
18.8.3	Security and Medical Treatment	159
18.8.4	Laboratory and Geology Sample Preparation	159
18.8.5	Process Plant Change-rooms and Mess Hall	160
18.8.6	Process Plant Ancillary Buildings and Facilities	160

18.8.7	Warehouse.....	160
18.8.8	Permanent Camp	160
18.9	MINE SITE SURFACE INFRASTRUCTURE.....	160
18.10	TAILINGS AND WASTE ROCK MANAGEMENT	161
18.10.1	Existing Tailings Impoundment.....	161
18.10.2	TSF Expansion Options	162
18.10.3	Proposed TSF Expansion	162
18.10.4	Tailings Characteristics.....	164
18.10.5	Site Characteristics.....	165
18.10.6	Geotechnical Considerations.....	167
18.10.7	TSF Design (Site ‘D’).....	169
18.10.8	Closure	172
19.0	MARKET STUDIES AND CONTRACTS.....	174
19.1	GOLD SALES	174
19.2	MATERIAL CONTRACTS	174
20.0	ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT	175
20.1	ENVIRONMENTAL STUDIES AND ISSUES	175
20.1.1	Environmental Conditions	175
20.1.2	Environmental Impacts	176
20.1.3	Environmental and Social Management	176
20.2	WASTE AND WATER MANAGEMENT	177
20.3	PERMITTING REQUIREMENTS	177
20.3.1	Regulatory Framework	177
20.3.2	Required Environmental Permits	180
20.4	SOCIAL AND COMMUNITY ASPECTS.....	181
20.4.1	Social Conditions	181
20.4.2	Social Development Initiatives	181
20.4.3	Health and Safety	182
20.5	MINE CLOSURE REQUIREMENTS	183
21.0	CAPITAL AND OPERATING COSTS.....	184
21.1	CAPITAL COSTS	184
21.1.1	Mining Capital Cost	184
21.1.2	Plant Expansion Capital Expenditure.....	184
21.1.3	Sustaining Capital	186
21.1.4	Closure/Reclamation Costs	186
21.2	OPERATING COSTS.....	186
21.2.1	Mine Operating Costs	186
21.2.2	Processing Operating Costs	187
21.2.3	General and Administrative Costs	187
22.0	ECONOMIC ANALYSIS.....	188
22.1	BASIS OF EVALUATION	188
22.2	MACRO-ECONOMIC ASSUMPTIONS	188

22.2.1	Exchange Rate and Inflation	188
22.2.2	Expected Metal Prices.....	188
22.2.3	Weighted Average Cost of Capital	189
22.2.4	Taxation Regime	190
22.2.5	Royalty	190
22.2.6	Marketing Costs	190
22.3	TECHNICAL ASSUMPTIONS.....	190
22.3.1	Mine Production Schedule	190
22.3.2	Stockpiling	192
22.3.3	Processing Schedule.....	192
22.3.4	Operating Costs.....	193
22.3.5	Capital Costs	193
22.3.6	Project Cash Flow	194
22.4	BASE CASE EVALUATION.....	195
22.5	SENSITIVITY STUDY	197
22.5.1	Sensitivity to Revenue Factors, Operating and Capital Costs.....	197
22.5.2	Gold Price and Discount Rate	198
22.6	CONCLUSION.....	199
23.0	ADJACENT PROPERTIES	200
23.1	LA VICTORIA AND TOMI MINES	201
23.2	ISIDORA MINE (MINA CHILE).....	201
23.3	COLUMBIA MINE.....	202
24.0	OTHER RELEVANT DATA AND INFORMATION	203
24.1	PROJECT SCHEDULE	203
24.2	PROJECT IMPLEMENTATION	203
25.0	INTERPRETATION AND CONCLUSION	205
26.0	RECOMMENDATIONS.....	206
26.1	GEOLOGY	206
26.2	GEOTECHNICAL.....	206
26.3	MINING - WASTE STORAGE.....	207
26.4	METALLURGY	208
26.5	SCHEDULE.....	208
26.6	POWER SUPPLY	208
26.7	CAPITAL ESTIMATE.....	208
26.8	BUDGET	209
27.0	DATE AND SIGNATURE PAGE	210
28.0	REFERENCES.....	211
29.0	CERTIFICATES.....	214

List of Tables

	Page
Table 1.1 Choco 10 Mineral Resource Estimate.....	5
Table 1.2 Incredible 6 Mineral Resource Estimate	5
Table 1.3 Choco 10 Mineral Reserves (December 31, 2010)	6
Table 1.4 Incredible 6 Mineral Reserves (December 31, 2010)	6
Table 1.5 Gold Price Forecasts	12
Table 1.6 Life-of-Mine Cash Flow Summary	15
Table 1.7 Base Case Cash Flow Evaluation.....	17
Table 1.8 Sensitivity to Gold Price and Discount Rate.....	18
Table 2.1 List of Abbreviations	24
Table 10.1 Choco 10 Deposits – Drill Summary	47
Table 10.2 2007 to 2009 – Drill Summary	48
Table 10.3 Drill Statistics.....	49
Table 10.4 Rusoro Geological Procedures.....	51
Table 11.1 Rusoro Sample Preparation and Assay Process	56
Table 11.2 Rusoro Quality Control Protocols.....	56
Table 11.3 Rusoro QC Sample Failure Criteria	57
Table 11.4 Rusoro QC Failure Action Plan	57
Table 11.5 Rusoro Drill Hole Sample Laboratories.....	58
Table 13.1 Samples submitted to McClelland for testing in 1995.....	68
Table 13.2 Composite Samples used in the Amdel Test Program	68
Table 13.3 Summary of Grinding Testwork Results	69
Table 13.4 Summary of Gravity Testwork Results.....	70
Table 13.5 Summary of McClelland Cyanidation Testwork Results	70
Table 13.6 Summary of Amdel Cyanidation Testwork Results.....	71
Table 13.7 Summary of Lakefield Cyanidation Bottle Roll Testwork Results	72
Table 13.8 Summary of Samples Used in 2008-2010 Testwork	73
Table 13.9 Composite Sample Identifiers	74
Table 13.10 Head Assay Comparison of New Composites Tested	76
Table 13.11 Hardness and Bulk SG Data on New Composites	76
Table 13.12 Head Assay Comparison of Less Extractable Intervals	76
Table 13.13 Gravity Response of Less Extractable Intervals	77

Table 13.14	Summary of Diagnostic Leach Findings.....	78
Table 13.15	Baseline Grind-Sensitivity Data for Leach Evaluation.....	79
Table 13.16	Effect of Oxidative Leach Enhancements at 74 µm	80
Table 13.17	Effect of Standard Carbon Additions at 74 µm.....	81
Table 13.18	Typical CIP Solution Assays	82
Table 13.19	Average Whole Ore Leach Responses per Sample.....	83
Table 13.20	Bond Ball Mill Work and Abrasion Indices	83
Table 13.21	Summary of SMC Evaluations	83
Table 13.22	Sample Identification	85
Table 13.23	Sample Description and Source Location	85
Table 13.24	JK Tech Drop-Weight Test	85
Table 13.25	SMC Test Results.....	86
Table 13.26	Bond Rod Mill Work Index Determinations.....	86
Table 13.27	Bond Abrasion Index Determinations.....	87
Table 13.28	Carbon-in Leach and Cyanidation Leach Tests	87
Table 13.29	Rheology Testwork	88
Table 13.30	Thickener Design Testwork	88
Table 14.1	Gold Fields Mineral Resource Estimates as of September 30, 2007	90
Table 14.2	Deposit Basic Statistics.....	93
Table 14.3	Variogram Parameters.....	94
Table 14.4	Block Model Geometry.....	96
Table 14.5	Block Model Attributes.....	96
Table 14.6	Block Model Parameters.....	97
Table 14.7	Rosika Measured Mineral Resource	100
Table 14.8	Rosika Indicated Mineral Resource	100
Table 14.9	Rosika Inferred Mineral Resource	100
Table 14.10	Coacia Measured Mineral Resource	101
Table 14.11	Coacia Indicated Mineral Resource	101
Table 14.12	Coacia Inferred Mineral Resource	101
Table 14.13	Pisolita Measured Mineral Resource	102
Table 14.14	Pisolita Indicated Mineral Resource	102
Table 14.15	Pisolita Inferred Mineral Resource	103
Table 14.16	VBK Measured Mineral Resource	103

Table 14.17	VBK Indicated Mineral Resource	103
Table 14.18	VBK Inferred Mineral Resource	104
Table 14.19	Choco 10 Mineral Resource Estimates	104
Table 14.20	Choco 10 Mineral Resource Comparison	105
Table 14.21	Previous Incredible 6 Mineral Resource Estimate, September 2007	106
Table 14.22	Incredible 6 Deposit Basic Statistics	108
Table 14.23	Incredible 6 Variogram Parameters	111
Table 14.24	Ingrid-Olga-Christina Block Model Geometry	111
Table 14.25	Culebra Block Model Geometry	112
Table 14.26	Block Model Attributes	112
Table 14.27	Incredible 6 Block Model Parameters	113
Table 14.28	Ingrid Measured and Indicated Mineral Resource	114
Table 14.29	Ingrid Inferred Mineral Resource	114
Table 14.30	Christina Measured and Inferred Mineral Resource	115
Table 14.31	Christina Inferred Mineral Resource	115
Table 14.32	Olga Measured and Inferred Mineral Resource	116
Table 14.33	Olga Inferred Mineral Resource	116
Table 14.34	Culebra Measured and Indicated Mineral Resource	117
Table 14.35	Culebra Inferred Mineral Resource	117
Table 14.36	Incredible 6 Mineral Resource at 0.5 g/t Au Cut-off	118
Table 14.37	Incredible 6 Mineral Resource Comparison	118
Table 15.1	Choco 10 Mineral Reserves (December 31, 2010)	119
Table 15.2	Incredible 6 Mineral Reserves (December 31, 2010)	119
Table 16.1	Parameters used for the Choco 10 Whittle Pit Optimization	121
Table 16.2	Parameters used for the Incredible 6 Whittle Pit Optimization	121
Table 16.3	Parameters used for the Culebra Whittle Pit Optimization	121
Table 16.4	Whittle Results – Choco 10	123
Table 16.5	Whittle Results – Incredible 6	124
Table 16.6	Whittle Results – Culebra	125
Table 16.7	Mine Production Schedule	126
Table 16.8	Annual Mine Equipment Requirements	129
Table 16.9	Equipment Replacement Requirements	130
Table 17.1	Process Design Criteria Summary	141

Table 17.2	Ore Deposit Resource Tonnage Ratio.....	145
Table 17.3	Primary Ore Comminution Parameters.....	146
Table 17.4	Grinding Mill Design Criteria.....	147
Table 17.5	Pre-Leach Thickening Design Criteria.....	149
Table 17.6	Leaching and Adsorption Design Criteria.....	150
Table 20.1	Increible Impacts and Mitigation	176
Table 20.2	Applicable Legislation	178
Table 20.3	Permit Status for Project Changes, New Affection Licences, and/or Exploitation Certificates	180
Table 21.1	Project Capital Expenditure Summary.....	184
Table 21.2	Plant Expansion Capital Expenditure.....	185
Table 21.3	Plant Services Capital Expenditure	185
Table 21.4	Infrastructural Capital Expenditure.....	185
Table 21.5	Indirect and Other Capital Expenditure	185
Table 21.6	Process Operating Costs.....	187
Table 22.1	Gold Price Forecasts	188
Table 22.2	Life-of-Mine Cash Flow Summary.....	194
Table 22.3	Base Case Life of Mine Annual Cash Flow.....	196
Table 22.4	Base Case Cash Flow Evaluation.....	197
Table 22.5	Sensitivity to Gold Price and Discount Rate.....	198
Table 23.1	La Victoria and Tomi, Production	201
Table 23.2	Isidora Mine Mineral Resources, March 31, 2008.....	201
Table 23.3	Isidora Mine Mineral Reserve, March 31, 2008	201
Table 26.1	Project Development Budget to March 2013.....	209

List of Figures

	Page
Figure 1.1	Location of the Choco 10 Mine, Eastern Venezuela.....1
Figure 1.2	Project Implementation Schedule 10
Figure 1.3	Ore Mining Production Schedule.....13
Figure 1.4	LOM Processing Schedule and Grade Profile 14
Figure 1.5	Cash Operating Costs.....14
Figure 1.6	Capital Expenditures 15
Figure 1.7	Life-of-Mine Cash Flows.....16
Figure 1.8	Sensitivity to Capital, Operating Costs and Revenue 17
Figure 4.1	Location of the Choco 10 Mine, Venezuela.....27
Figure 4.2	El Callao District Showing Locations of Rusoro Concessions.....28
Figure 5.1	Plan Showing the Choco 10 Mining Areas and Mine Infrastructure 31
Figure 7.1	Regional Geology of the Northern Guyana Shield Showing Major Gold Deposits.....35
Figure 7.2	Regional Geology of the Choco 10 and Other Major Gold Deposits 35
Figure 7.3	Local Geology of the El Callao District.....37
Figure 7.4	Stratigraphic Sequence and Mineralization of the Choco 10 Deposit 38
Figure 7.5	Local Geology, Choco 10 39
Figure 10.1	Choco 10 Drill Hole Collar Locations 48
Figure 10.2	Increible 6 Drill Hole Collar Locations 50
Figure 11.1	Sample Flowsheet for Choco 10 Drill Samples 55
Figure 11.2	Sample Consignment Form.....60
Figure 12.1	Drill Hole Data Entry 63
Figure 12.2	Assay Laboratory Sif File 64
Figure 12.3	Data Shed - Data Validation 65
Figure 12.4	Assay Swapper 65
Figure 13.1	Diagnostic Leach Residue Grades 78
Figure 13.2	Grind Sensitivity of Whole Ore Leaching 79
Figure 13.3	Labile Sulphide Oxidation Effects 80
Figure 13.4	Effects of Carbon Additions.....81
Figure 13.5	Un-weighted Average Performance Summary 82
Figure 14.1	VBK+RCP Geological Wireframe Model 92

Figure 14.2	Rosika and Coacia Histogram and Frequency Distribution Graphs	93
Figure 14.3	Pisolita and VBK Histogram and Frequency Distribution Graphs	93
Figure 14.4	Semi-Variograms for RCPVBK.....	95
Figure 14.5	Incredible 6 Deposit Wireframe Solids.....	107
Figure 14.6	Ingrid and Christina Histogram and Frequency Distribution Graphs	108
Figure 14.7	Olga and Culebra Histogram and Frequency Distribution Graphs	109
Figure 14.8	Semi-Variograms for Incredible 6 Deposits	110
Figure 16.1	Life of Mine Pit Design – Choco 10 (VBK-Rosika-Coacia-Pisolita).....	127
Figure 16.2	Life of Mine Pit Design – Incredible 6 (Culebra-Cristina-Elisa).....	128
Figure 17.1	Schematic Flow Diagram of the Existing Plant (5,000 t/d)	133
Figure 17.2	Historical Production from the Choco 10 Plant	135
Figure 17.3	Flowsheet for Expansion to 20,000 t/d (5,000 + 15,000 t/d)	137
Figure 17.4	Proposed Site Layout	138
Figure 18.1	Schematic Diagram of Existing TSF Layout (not to scale)	161
Figure 18.2	Proposed TSF Layout (Site ‘D’)	163
Figure 18.3	Typical Cross-Section of Embankment	170
Figure 18.4	TSF General Arrangement - Stage 7.....	171
Figure 22.1	Gold Price 2001-2010	189
Figure 22.2	Real Return on US Long Bonds 2003-2010	189
Figure 22.3	Ore Mining Production Schedule.....	191
Figure 22.4	Ore and Waste Mining Production Schedule	191
Figure 22.5	Ore Stockpiles – Closing Balances	192
Figure 22.6	LOM Processing Schedule and Grade Profile	192
Figure 22.7	Cash Operating Costs.....	193
Figure 22.8	Capital Expenditures	194
Figure 22.9	Life-of-Mine Cash Flows.....	195
Figure 22.10	Sensitivity to Capital, Operating Costs and Revenue	198
Figure 23.1	El Callao Gold District - Adjacent Properties Map	200
Figure 24.1	Project Implementation Schedule	204

1.0 SUMMARY

Micon International Limited (Micon) was retained in August, 2009 by Rusoro Mining Ltd. (Rusoro) to complete a feasibility study for an expansion of the existing processing facility to permit treatment of 20,000 t/d of mainly hard rock from the Choco 10 and Incredible 6 deposits. Exploration prior to 2009 had identified additional mineralised deposits and, in 2008, a preliminary assessment was conducted which examined the possibility of increasing the 5,000 t/d mill capacity (Buchanan et. al., 2009). A number of options were investigated and increasing the total plant throughput to 20,000 t/d was found to be the most economically viable.

This feasibility study evaluates the construction of additional processing facilities to treat material from the existing Choco 10 mine and the nearby Incredible 6 property as an expansion of the existing Choco 10 plant. With the exception of the construction of a new tailings storage facility, the expanded facilities will be essentially located within the site of the existing Choco 10 operation and represent a brownfields expansion.

Figure 1.1
Location of the Choco 10 Mine, Eastern Venezuela



Rusoro's operations are located in the vicinity of El Callao and El Dorado, Bolivar State, in eastern Venezuela. These comprise two operating gold mines, Choco 10 and Isidora, and 10 exploration projects (including development and exploration around the mines). Output from both the Choco 10 and Isidora mines, near El Callao, is processed through the Choco mill. (Ore from Isidora has been processed through the Choco mill since late 2008 and reserves are expected to be exhausted soon). Through Promotora Minera de Guayana S.A., (PMG), the Venezuelan operating company, Rusoro owns and operates the Choco mill and has an ownership interest of 95% in the Choco 10 mine. Production from the Isidora mine is subject

to a joint venture with the Venezuelan government. The Incredible 6 gold property is under development and is located 4 km northeast of Choco 10. It is 100% owned by Rusoro, through its subsidiary, General Mining de Guyana, C.A. (General Mining).

In addition to the preparation of the preliminary assessment referred to above, Micon was previously retained by Rusoro to audit the resource estimate for Choco 10 and to prepare a resource estimate for Incredible 6. The respective Technical Reports discussing these estimates are:

- Technical Report on the PMG (Gold Fields) Choco 10 Concession and Mine, Estado Bolivar, Venezuela, dated November 21, 2007. (Leader et al., 2007).
- Technical Report on the Incredible 6 Property, Bolivar State, Venezuela, dated November 14, 2007, revised and updated February 14, 2008. (Laudrum et al., 2008).

1.1 INTRODUCTION

The Choco 10 mine commenced production in August, 2005. Current operations consist of open pit mining and a processing plant comprising conventional comminution and carbon-in-pulp processing. The Choco 10 mine uses typical open pit mining methods of drilling, blasting, loading and hauling. Two pits are currently being operated within the Choco 10 concession, Pisolita and Rosika-Coacia. The pits are located 2-3 km from the main plant.

The Choco 10 concession has a Certificate of Exploitation granted to PMG for an area of 2,124.53 ha and a term running to December 25, 2025. The Choco 4 concession has a Certificate of Exploitation granted to PMG for an area of 1,458.12 ha and a term running to December 25, 2025.

On January 14, 2009, Rusoro announced that for Incredible 6, “approval for the Certificate of Exploitation was published in Venezuela’s Official Gazette on December 23, 2008”. On receipt of the Exploitation Certificate, application will be made for the Permit to Affect Natural Resources. The Ministry of the People’s Power for the Environment (MinAmb) issued the authorization to affect Natural Resources for the Incredible 6 deposits on November 1, 2010. This permit was the final step in the permitting process toward the commencement of mining activities at the Incredible 6 deposits.

1.2 PROPERTY DESCRIPTION

Choco 10 and Incredible 6 deposits lie in an area of low hills between elevations 200 masl and 300 masl. The area is partly savannah and partly tropical forest. The soil is nutrient-poor and the present land use is confined to rural cultivation, cattle ranching and *minero* (small mining) activities.

The climate at Choco 10 and Incredible 6 deposits is tropical, with temperatures averaging around 25.7°C and humidity ranging from 76% to 82%. The estimated mean annual precipitation at El Callao is 1,325 mm. The heaviest rainfall typically occurs during the months from June to August.

A paved secondary road passes through the concessions in Choco 10 and provides access to the town of El Callao, about 15 km to the east of the concessions. Access to Incredible 6 from El Callao is by paved road, except for the last 2 to 3 km which are unpaved. El Callao has a population of approximately 25,000 and is the centre of population in the area. It is an historic gold mining centre and a number of present and past producing mines are located nearby. The Incredible 6 deposit is connected to the Choco mine and mill by a 6 km gravel road.

The main highway, between the regional centre of Puerto Ordaz, a major city on the Orinoco River 190 km to the north, and other gold mining centres of El Dorado and Kilometre 88, and the Brazilian border to the south, passes through El Callao.

A major high voltage 400 kV electrical supply line carrying power from the Guri dam, and destined for Brazil, passes near El Callao. A substation about 5 km from the Choco concessions supplies power to El Callao and the Choco 10 operation.

A rain-dependent reservoir supplies water for use at the mine, and this is supplemented by a water well field. Another potential water supply is from the Yuruarí River, which passes north of the concessions, approximately 8 km distant, and from which other local mines and the El Callao municipality draw water.

Most of the personnel working at the mine live in and around El Callao. The population is familiar with mining and can provide the majority of the labour requirement.

1.3 HISTORY

Gold mining in the El Callao district dates back to the early Spanish Conquest in the 1500s in the search to find El Dorado.

On October 11, 2007, Rusoro acquired all of the Venezuelan assets of Gold Fields Limited (Gold Fields), including the Choco 10 gold mine. Prior to Rusoro's acquisition, Gold Fields held the rights to Choco 10 and other Venezuelan assets that were acquired through a plan of arrangement with Bolivar Gold Corp. (Bolivar) on February 28, 2006. Under the plan of arrangement Gold Fields acquired all of the outstanding securities in Bolivar which it did not already hold. Gold Fields owned its interest in the Choco 10 mine through its holding in Promotora Minera de Guayana S.A. PMG remains the operating company for Choco 10 and is owned 95% by Promotora Minera de Venezuela SA (Promiven) and 5% by CVG Minervén C.A. Promiven is wholly owned by Rusoro through Venezuela Holdings (BVI) Ltd. and Carisma Corporation AVV.

There has been a significant amount of shallow artisanal gold mining within the Incredible 6 concession in the past. There are no records of when this work commenced or the total production to date. Through a wholly-owned subsidiary, Rusoro acquired the Incredible 6 property in 2004 and has been actively exploring it since that time.

1.4 GEOLOGY AND MINERALIZATION

The Choco 10 and Incredible 6 mineral concessions are located within the Guyana Shield, which occupies the northern part of the Amazon Craton between the Amazon and Orinoco river basins. The geology of the Guyana Shield as a whole is poorly known, reflecting minimal development and limited access, as well as intense tropical weathering and cover across this large area. It has been subdivided into three major geological entities:

- i) Archaean rocks of the Imataca Complex.
- ii) Palaeoproterozoic Trans-Amazonian granite-greenstone belts.
- iii) Palaeoproterozoic sedimentary and igneous rocks of the Roraima Group, Uatuma Group and the Avanavero Suite. The majority of gold mineralization is hosted by the greenstone belts.

The gold mineralization at Choco 10 and Incredible 6 is typical of Archaean-Proterozoic orogenic gold deposits. The deposits are hosted in an early Proterozoic sequence of the Pastora Greenstone Belt of the Guyana Shield. The stratigraphy comprises a tholeiitic to calc-alkaline volcanic package, overlain by volcanoclastic and epiclastic rocks intruded by gabbroic sills. The rock package has been subjected to intense tropical weathering and mineralization is dominantly structurally controlled. High-grade gold mineralization occurs with carbonate, pyrite, silicification and quartz-veining in lower-strain zones typically associated with crenulations, folding and chaotic foliations.

1.5 EXPLORATION AND OTHER PROGRAMS

Exploration activities at the Choco concessions have included drilling, trench excavation, surveying and mapping. Numerous studies were also prepared including Petrographic studies, and Structural studies. The exploration work carried out at the mine site by Gold Fields from March 2006 included, primarily, infill drilling designed to advance resource classifications plus in-pit (to depth) and step-out drilling to expand resources. Since Rusoro acquired the Choco 10 operations, they have continued to undertake a series of drilling programs to delineate potential ore extensions both horizontally and down the dip of each of the deposits as well as continue drill definition of the known mineralized zones to raise each of the deposit's Mineral Resource categories.

Between 1991 and 1994, General Mining carried out an exploration program on Incredible 6 that consisted of soil sampling, trenching and diamond drilling. Starting in 1994, Golden Star conducted exploration program on Incredible 6 which included completing a soil and auger program, trenching, geophysics, geological mapping, and a drilling program. Their advanced work was concentrated on the east-west trending quartz veins and shears in the Nuevo Rosario area.

In 2004, Rusoro began exploration and drilling on the concession. Since 2004, Rusoro has concentrated most of its efforts on a series of extensive RC and diamond drilling programs. They have also completed small programs of line cutting, soil sampling, and petrographic

work. Surface surveying included a detailed topographic survey over the main Ingrid-Elisa area allowing for the preparation of an updated 2-m contour topographic map.

1.6 MINERAL RESOURCE ESTIMATE

The Micon mineral resource reports on the Choco 10 (Makepeace, 2010), and Incredible 6 deposits (Laudrum et al., 2008 and Makepeace 2010) form the basis for the feasibility study which is the subject of this report.

The mineral resources for Choco 10, as at December 31, 2009, are summarized in Table 1.1 and are reproduced from Makepeace, 2010. The resources are quoted separately for the four deposits on the property: Rosika, Coacia, Pisolita (RCP) and Villa Balazo-Karolina (VBK).

Table 1.1
Choco 10 Mineral Resource Estimate
Above 0.50 g/t Au cut-off, as of December 31, 2009

Deposit	Measured			Indicated			Total M + I			Inferred		
	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Rosika	3.1	1.94	192	15.2	2.19	1,074	18.3	2.15	1,267	12.2	1.55	607
Coacia	0.1	1.74	8	1.2	1.52	58	1.3	1.55	66	8.3	1.68	449
Pisolita	0.2	1.63	11	5.1	1.57	260	5.3	1.58	271	18.7	1.15	690
VBK	31.3	2.20	2,214	83.6	1.67	4,488	114.9	1.81	6,702	20.0	1.67	1,076
TOTAL	34.7	2.17	2,425	105.2	1.74	5,880	139.9	1.85	8,305	59.2	1.48	2,821

- 1) It cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.
- 2) Mineral resources which are not mineral reserves do not have demonstrated economic viability.

The mineral resources for Incredible 6 are summarized in Table 1.2. The resources are quoted separately for the four deposits: Culebra, Cristina, Elisa-Ingrid and Olga-Enoc. These estimates are based on data available at December 31, 2009.

Table 1.2
Incredible 6 Mineral Resource Estimate
Above 0.50 g/t Au cut-off, as of December 31, 2009

Deposit	Measured			Indicated			Total M + I			Inferred		
	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Elisa/Ingrid	0.01	2.01	0.8	13.70	2.10	927.0	13.72	2.10	927.8	8.62	1.52	420.2
Christina	0.25	2.02	16.2	1.87	1.47	88.3	2.12	1.53	104.5	0.02	0.92	0.6
Olga/Enoc	0.02	2.58	1.4	0.57	1.21	22.1	0.59	1.25	23.5	0.23	0.88	6.5
Culebra	0.18	2.35	13.7	5.83	1.62	302.9	6.01	1.64	316.6	0.71	1.34	30.3
TOTAL	0.46	2.17	32.1	21.97	1.90	1,340.3	22.43	1.90	1,372.4	9.58	1.49	457.7

- 1) It cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.
- 2) Mineral resources which are not mineral reserves do not have demonstrated economic viability.

These resource estimates include inferred resources, which are not included in the open pit designs and production schedules for the feasibility study. There is no assurance that further

exploration will result in upgrading any of these inferred resources to the measured or indicated category.

Two other small deposits, Capia and Cerro Azul that were included in the preliminary assessment are not material and have not been included in this feasibility study.

1.7 MINERAL RESERVE ESTIMATE

The feasibility study considered mining from the Choco 10 and the Increible 6 deposits at a combined rate of 20,000 t/d or 7.3 Mt/y of mill feed. New resource block models generated by Micon were used as the basis for the open pit designs.

As of December 31, 2010, the mineral reserves for the Choco 10 and Increible 6 deposits are as summarized in Table 1.3 and Table 1.4 respectively.

Table 1.3
Choco 10 Mineral Reserves (December 31, 2010)

Deposit	COG	Proven			Probable			Total P+P		
		(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Rosika	0.4	1.95	1.88	118	8.15	2.10	549	10.10	2.05	667
Coacia	0.4	0.18	1.28	8	0.61	1.39	27	0.79	1.36	35
Pisolita	0.4	0.07	1.35	3	1.26	1.91	78	1.33	1.88	81
VBK	0.4	28.66	1.82	1,676	45.77	1.36	2,005	74.43	1.54	3,681
TOTAL	0.4	30.86	1.82	1,804	55.79	1.48	2,659	86.65	1.60	4,463

Table 1.4
Increible 6 Mineral Reserves (December 31, 2010)

Deposit	COG	Proven			Probable			Total P+P		
		(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Culebra	0.4	0.17	2.23	12	3.49	1.69	190	3.65	1.72	202
Christina	0.4	0.09	2.40	7	0.67	1.65	36	0.76	1.74	43
Elisa/Ingrid	0.4	0.01	2.02	1	4.81	2.03	314	4.82	2.03	315
Olga/Enoc	0.4	0.00	0.00	0	0.00	0	0	0.00	0.00	0
TOTAL	0.4	0.27	2.28	20	8.97	1.87	540	9.24	1.88	559

Mining losses of 1% were assumed, and dilution factors of 11% and 14% were applied to oxide ore and transition/fresh ore respectively. Ore tonnes were diluted by an average grade of 0.10 g/t. These mineral reserves are included within the mineral resources stated in Section 1.6. The mine design work supporting these reserves is described more fully in Section 16.0.

1.8 MINING METHODS

The Rosika, Coacia and Pisolita deposits have been mined for 6 years. The current mill has a capacity of 5,000 t/d. It is proposed to increase the plant capacity to 20,000 t/d. This increase in capacity is based on continuing to mine the three current pits, bringing the relatively large VBK open pit on line as well as developing the Incredible 6 deposits.

Mining will be carried out by a contractor, most likely using a fleet of up to 25 240-t trucks with suitable hydraulic shovels (3 x 28m³), wheel loaders, drills and supporting equipment.

1.9 METALLURGICAL TESTWORK

Metallurgical testwork carried out prior to 2008 was undertaken to support the mineral resource estimate for Choco 10 and the selection of the processing flowsheet for the existing mill at Choco 10. The existing milling facility is rated at 5,400 t/d on saprolitic ore but is expected to only be able to treat 5,000 t/d when receiving more competent material.

In 2008, metallurgical testing of the competent fresh rock was conducted by Process Research Associates Ltd. (PRA), Richmond, British Columbia. Cyanide leach tests for the Choco 10 composite samples indicated that the material responded well, with gold recoveries generally over 84%. These results demonstrated that the current process treatment, using a grind size of approximately 80% passing (P₈₀) 74 µm and 48-h leach time, is applicable to future mined material and should result in gold recoveries of around 90%.

The feasibility metallurgical testwork program concentrated on refining the conditions as determined in the 2009 Preliminary Assessment report. Additionally work was conducted to determine the potential of gold recovery using flotation and intensive cyanide leaching. The bulk of this work was carried out by PRA.

To assess gravity concentration potential, testing was conducted using both Knelson and Falcon centrifugal type concentrators. The resulting concentrate produced represented a fairly high proportion of the total mass which required intensive leaching. The gold grades in the main stream were too high to discard and would, therefore, also require leaching, negating to a large extent the initial gravity separation step.

The use of flotation produced a 0.3 g/t Au tailing, however leaching of the flotation concentrate to enable production of a low grade residue was not achieved, although more testwork is recommended to prove/disprove this. On the basis that the resulting circuit would involve new technology (for the existing operation) and a more complex flowsheet with questionable benefits, it was decided not to continue this approach.

Leach testing at P₈₀ grinds of 55, 74, 100 and 150 µm, confirmed that the baseline of 74 µm was optimum as residue grades increased from 0.41, 0.42, 0.50 and 0.61 g/t respectively. Extensive testing was carried out to improve the leach kinetics as leach times for optimum recovery tended to be lengthy; these tests involved the use of lead nitrate and oxygen injection. Some minor improvement was achieved with oxygen.

A circuit modification from the current, conventional plant leach/CIP circuit was investigated. This modification commenced carbon adsorption after approximately half of the total leach cycle instead of at the end of the leach period. This hybrid configuration, although requiring a higher circuit carbon inventory, gave improved recoveries possibly because of pregnant robbing constituents in some of the ore types. It was decided that the plant circuit would consist of a hybrid leach/CIP circuit with a total retention time of 48 h whilst maintaining a cyanide concentration of 0.5 g/l. The plant would also be designed to include provision for oxygen injection into the leach tanks.

Testwork to determine semi autogenous grinding (SAG) milling characteristics were carried out at the ALS Ammtec Limited (Ammtec) metallurgical testing facilities in Western Australia. Fresh samples, which attempted to duplicate those previously tested at PRA, were sent from Venezuela.

JK Drop Weight tests were carried out on one sample from Rosika pit, and SAG mill comminution tests were carried out on all 7 samples.

The comminution testing indicated that to achieve a leach plant particle size with P_{80} of 74 μm and at a feed rate of 694 t/h (expanded circuit only) requires a SAG mill 9.75 m diameter (inside shell) and 4.26m effective grinding length; and a ball mill of 7.16 m diameter by 11.28 m long. The balance of the plant throughput would be milled in the existing circuit at 232 t/h.

Rheology testwork on a sample from the VBK deposit demonstrated that the slurry viscosity was low at natural and high pH but became fairly high at high slurry densities. However, the values are not sufficiently high to cause any pumping concerns.

Ammtec also conducted leach tests on the fresh ore samples to confirm that the conditions determined from the PRA work would achieve the design parameters of 89% gold recovery using a cyanide concentration of 0.5g/l with a 48-h leach/CIL retention time.

1.10 PROCESSING

The expanded facilities will consist of a new gyratory crusher which will be capable of treating 20,000 t/d. Crushed ore will then be transferred to a stockpile from which it will feed 15,000 t/d to the new milling circuit and 5,000 t/d to the existing facilities. The new plant will consist of a SAG mill which will operate in partial open circuit, with the trommel screen oversize being recycled through a pebble crusher. The trommel undersize will be combined with the ball mill discharge and pumped to a nest of classifying cyclones. The cyclone underflow returns to the ball mill and the overflow gravitates to a thickener for feed to the leach/CIL circuit.

The cyclone overflows from the two circuits will be thickened in a common unit, and the underflow split proportionally to feed the new leach/CIL circuit and existing CIP circuit.

The two Zadra elution circuits (new and existing) will be separate as the stripping cycle times will be different. Electrowinning and bullion production will occupy a new, secure building.

1.11 PROJECT INFRASTRUCTURE

A new power line from El Callao and attendant facilities will be required to cater for the increased power demand. There will also be a requirement for expansion in the capacity of some on-site infrastructure such as reagent mixing and office space; however, no separate facilities will be needed to support the new plant.

1.12 ENVIRONMENTAL AND SOCIAL ASPECTS

The Choco 10 project area falls within the Dry Tropical Forest bioclimatic zone of Venezuela. Historically, this zone has received an average annual rainfall of over 1,100 mm, with an average maximum rainfall during July of approximately 160 mm and a minimum average of 38.3 mm in March. The mean monthly temperature varies only slightly between 25°C and 27°C throughout the year.

The nearest community to the mine is the village of Choco while El Callao is the closest commercial centre. There are few indigenous people in the communities surrounding the Choco 4, Choco 10, and Incredible 6 concessions, and no fully indigenous communities. As part of its permit and corporate obligations, PMG consults with the local communities and has provided support and resources for the community and infrastructure development.

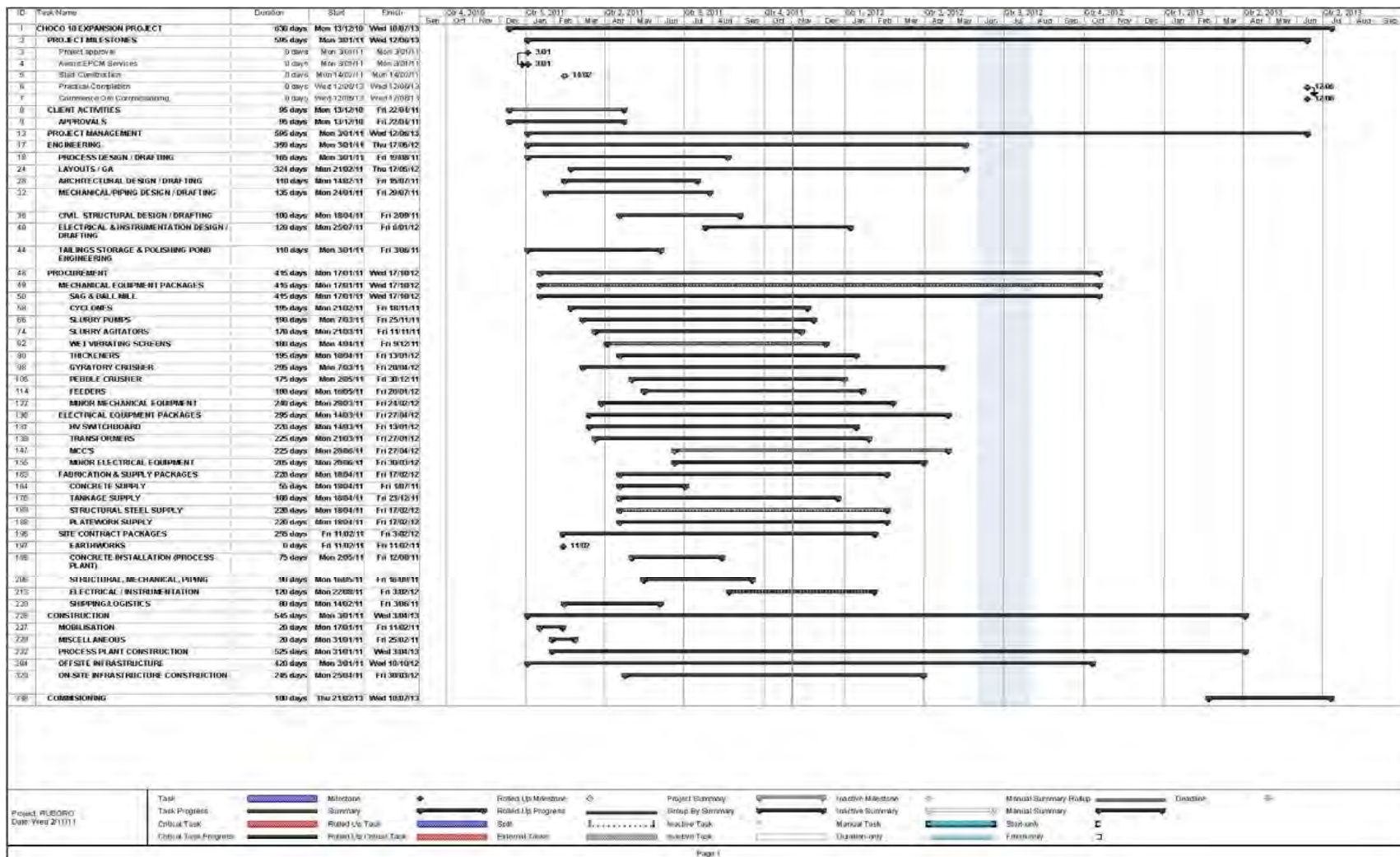
The exploitation and exploration permits are in place for Choco 10, Choco 4, and Incredible 6, and are kept current. Permitting is in progress for expansion of the project to include the new plant, waste rock dumps, tailings impoundment, and ancillary facilities.

The environmental management or supervision program for the Chocó mine has been expanded to include the environmental management plan requirements for the Incredible 6 mines. Requirements for the environmental management program incorporate conditions of the exploitation authorization and include compliance with all applicable legislation, avoiding protected areas, waste disposal, erosion control, hazardous materials storage, fire prevention, wildlife conservation, health, safety and security, effluents, water management, community consultation, among other items. Water and waste are being managed to prevent release of contaminants to the receiving environment. Mine reclamation is being carried out concurrently with operations and mine closure cost estimates have been included in the financial analysis.

1.13 PROJECT IMPLEMENTATION

Figure 1.2 shows a summary of the project implementation schedule based on a notional start date at the end of 2010, valid as of the effective date of this report.

Figure 1.2
Project Implementation Schedule



However, this schedule will need to be re-validated once PMG's approval for project implementation is given, and Micon notes that, as of the date of signing this report, slippage of one year had already occurred in the project implementation schedule.

1.14 CAPITAL COSTS

Capital expenditures for the expansion project are estimated to total \$310.8M, including \$152.8M processing, \$15.0M for services, \$41.9M infrastructure, \$14.8M for design growth, \$46.2M indirect costs and a contingency of \$40.1M, equivalent to almost 18% of the direct capital cost estimate. In addition, sustaining capital has been provided for in the amount of \$85.8M over the LOM period. A mine closure and rehabilitation provision of \$47.8 million is also provided for, including contributions of \$2M/y assumed over 15 years and final closure costs of \$17.8 million.

Working capital has been estimated to include 15 days product inventory comprising the mill, leaching circuit, carbon and elution inventories, and 15 days accounts receivable. Stores provision is for 60 days of consumables and spares inventory, less 30 days accounts payable. A balance of \$13.6M in working capital is assumed to be brought forward from the current operation.

1.15 OPERATING COSTS

Cash operating costs average \$22.73/t milled over the LOM period, including \$11.36/t mining, \$8.34/t processing and \$3.04/t general and administrative costs. The mining cost estimate includes a provision for increases in the contractors' unit rate to cover leasing charges in respect of the expanded mining fleet and its periodic replacement. This operating expense is in lieu of capital expenditure of approximately \$153 million for new mining equipment over the LOM period.

The initially high waste:ore ratios put pressure on operating margins in the first few years but, after Year 5, operating margins are increasingly robust over the remainder of the LOM.

1.16 ECONOMIC ANALYSIS

1.16.1 Basis of Evaluation

Micon has prepared its assessment of the Project on the basis of a discounted cash flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to establish the economic viability of the proposed expansion of open pit mining and processing for the production of gold from the Choco 10 and Incredible 6 deposits. In order to do this, the cash flow arising from the base case has been forecast,

enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

1.16.2 Macro-Economic Assumptions

1.16.2.1 Exchange Rate and Inflation

The project cash flow model, and all results derived from the model, are expressed in United States dollars (\$). Inputs to the cash flow model originating in Venezuelan Bolivar Fuerte (BsF) have been converted at the ruling rate¹. Constant, Q4-2010 money terms are used throughout, i.e., without provision for inflation.

1.16.2.2 Expected Metal Prices

The base case has been evaluated using trailing average metal prices to December 31, 2010, as shown in Table 1.5. In the light of upward momentum in the gold price, Micon has taken the 12-month average as its forecast price for the base case. As part of its sensitivity analysis, Micon also tested a range of prices 20% above and below the base case value.

Table 1.5
Gold Price Forecasts

Item	Gold (US\$/oz)
36-month trailing average to Dec-2010	1,023
24-month trailing average to Dec-2010	1,100
12-month trailing average to Dec-2010	1,225
1-month trailing average to Dec-2010	1,391

1.16.2.3 Weighted Average Cost of Capital

In order to find the NPV of the cash flows forecast for the project, an appropriate discount factor must be applied which represents the weighted average cost of capital (WACC) imposed on the project by the capital markets. The cash flow projections used for the valuation have been prepared on an all-equity basis. This being the case, WACC is equal to the market cost of equity.

The average real return on US long bonds is close to 2.0%, which is taken to be equal to the risk-free rate. Historically, the risk premium for equity has been estimated at 5.0%. The value of beta (β) for similar gold producers is observed to lie in the range 0.6 to 1.4 so, applying the Capital Asset Pricing Model (CAPM), the project's cost of equity will lie in the range of 5.0% to 9.0%. Accordingly, Micon has selected a discount rate of 7.0% as its base case estimate for the project's cost of equity and the economic results are presented for discount rates ranging from 5.0% to 9.0%.

¹ As of December, 2010 Venezuela's Central Bank applies a selling rate of BsF 4.30 per US\$. Prior to January 11, 2010, the selling rate was BsF 2.15 per US\$.

1.16.2.4 Taxation Regime

Venezuelan corporate income tax has been allowed for at the rate of 34%. Expansion project capital expenditure for the establishment of the new processing capacity, taken together with ongoing or sustaining capital, is assumed to be eligible for depreciation based on the unit of production method over the remaining life of the mine. The effective tax rate is thus approximately 28%.

1.16.2.5 Royalty

A royalty of 3.0% has been provided for in the cash flow model.

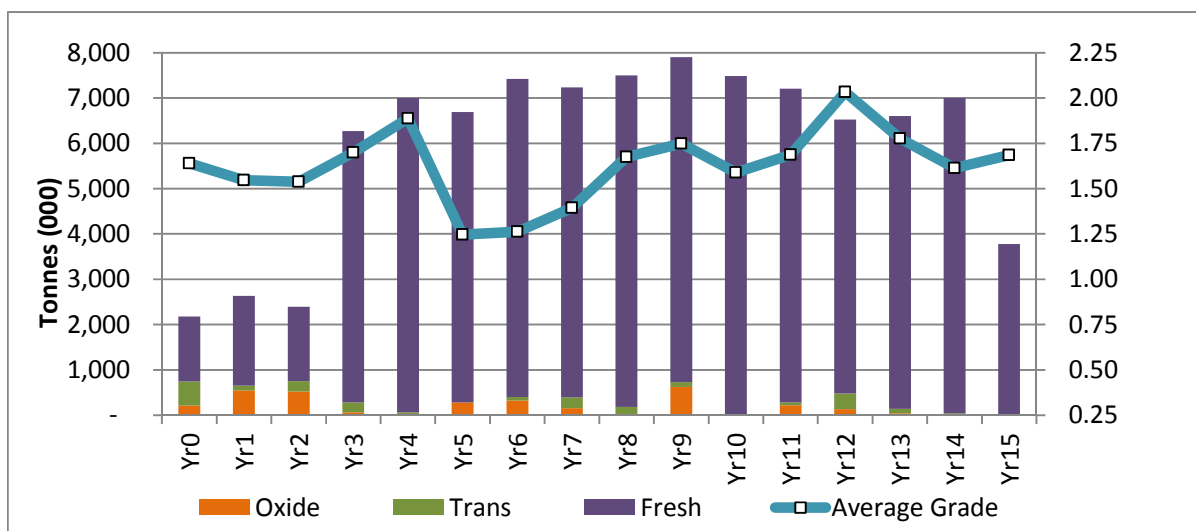
1.16.2.6 Marketing Costs

Costs relating to the refining and disposal of gold production take into account the market structure in Venezuela, in which 50% of the total sales are earmarked for sale to the Central Bank of Venezuela at the ruling exchange rate, and the remainder for export. For the purposes of this study, it is assumed that all cash flows are convertible to US dollars at the ruling exchange rate.

1.16.3 Mine Production Schedule

In Years 0 to 2, the current mill is fed at the rate of 5,000 t/d, comprising a mixture of oxide, transition and fresh ore types. During this period, waste mining ramps up to expose additional ore in anticipation of the expansion of milling capacity. After Year 3, the rate of ore mining increases, ROM tonnage being dominated by fresh ore, with remnant oxides being mined during stripping of the new open pits, including the Incredible area. Figure 1.3 shows the annual ore mining schedule.

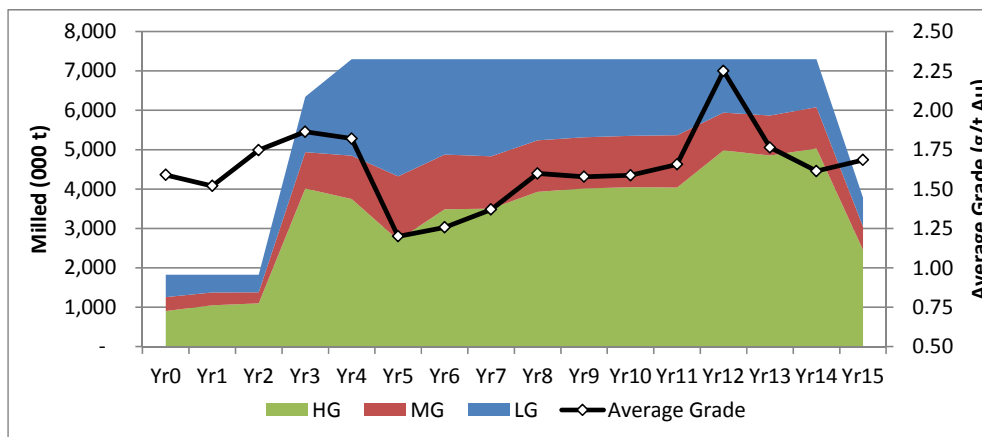
Figure 1.3
Ore Mining Production Schedule



1.16.3.1 Processing Schedule

Annual tonnage and grade of ore milled is shown in Figure 1.4.

Figure 1.4
LOM Processing Schedule and Grade Profile

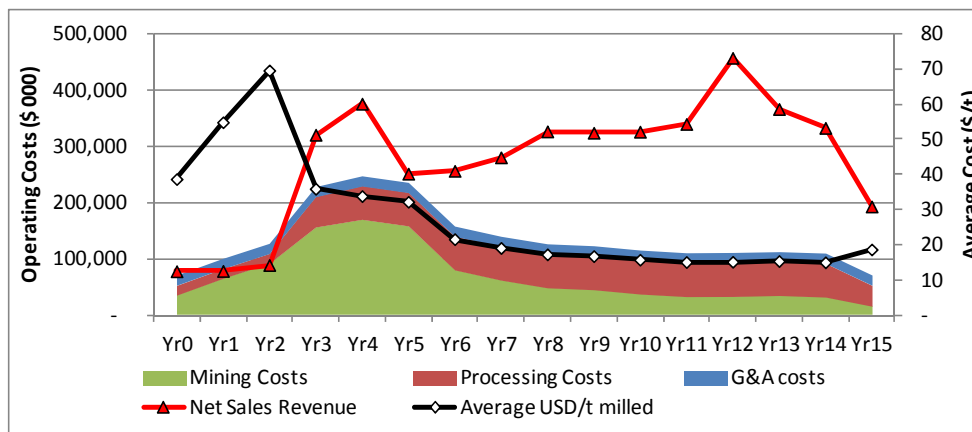


Following expansion of the process plant, in Year 3 significant quantities of low grade material are milled while the VBK open pit is being opened up. As the VBK pit develops, the proportion of high grade ore in the mill feed slowly increases, raising the average grade of ore milled from 1.25 g/t to around 1.75 g/t and then 2.25 g/t in Year 12.

1.16.3.2 Operating Costs

Cash operating costs average \$22.73/t milled over the LOM period, including \$11.36/t mining, \$8.34/t processing and \$3.04/t general and administrative costs. Figure 1.5 shows the operating expenditures over the LOM period.

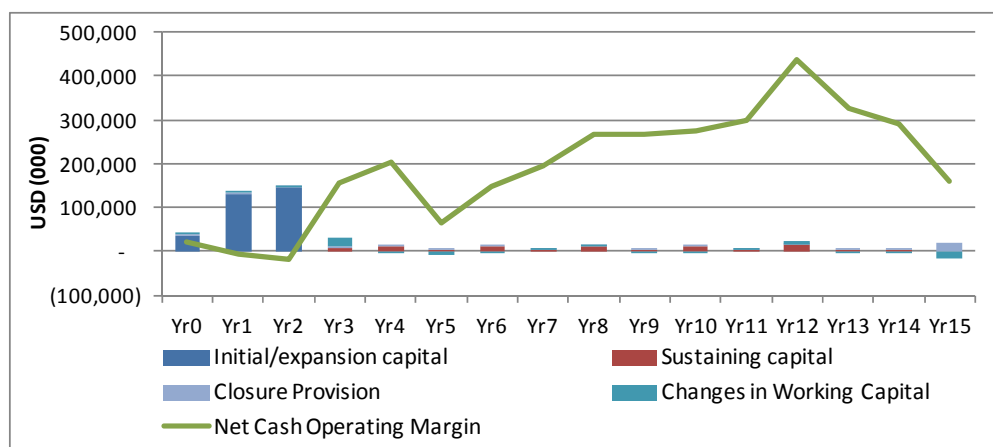
Figure 1.5
Cash Operating Costs



1.16.3.3 Capital Costs

Capital expenditures for the expansion project are estimated to total \$310.8M, including \$152.8M processing, \$15.0M for services, \$41.9M infrastructure, \$14.8M for design growth, \$46.2M indirect costs and a contingency of \$40.1M, equivalent to almost 18% of the direct capital cost estimate. In addition, sustaining capital has been provided for in the amount of \$85.8M over the LOM period, and a mine closure and rehabilitation provision of \$47.8 million is also provided for. Figure 1.6 compares annual capital expenditures over the preproduction and LOM periods with the project's cash operating margin.

Figure 1.6
Capital Expenditures



1.16.3.4 Project Cash Flow

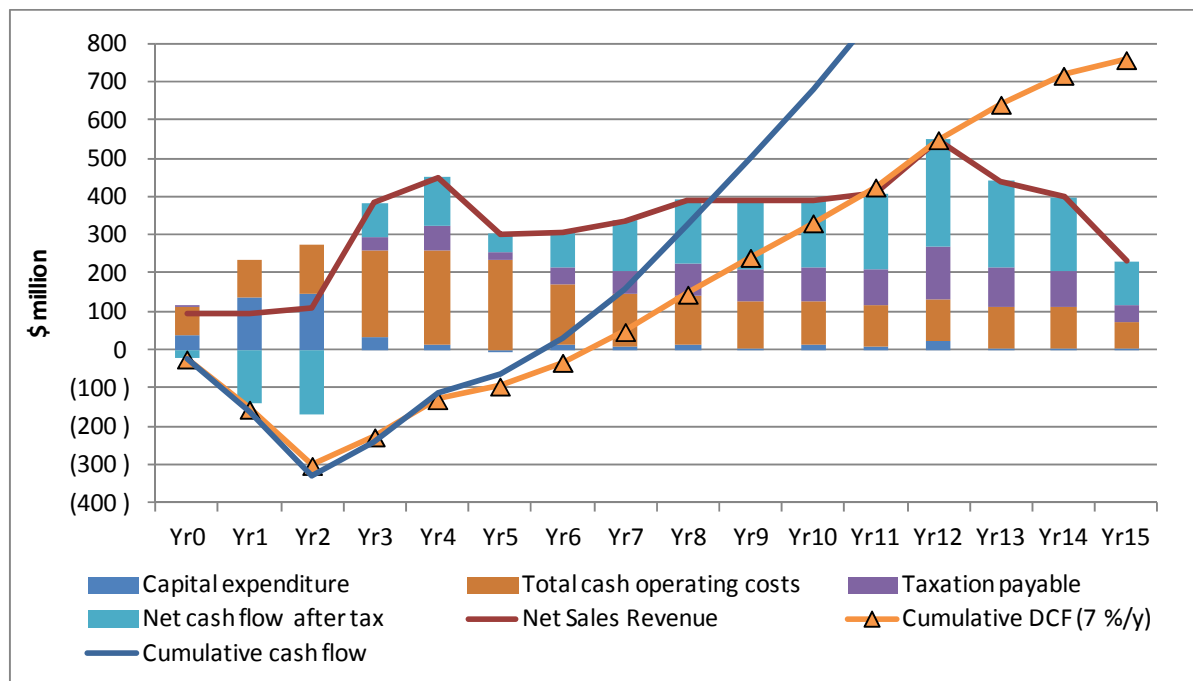
The LOM base case project cash flow is summarised in Table 1.6.

Table 1.6
Life-of-Mine Cash Flow Summary

	LOM Total (\$M)	\$/t Milled	\$/oz Gold
Gross Revenue	5,481.6	57.16	1,225.00
Selling Exp. & Royalty	(218.4)	(2.28)	(48.81)
Net revenue	5,263.2	54.88	1176.19
Mining costs	1,089.3	11.36	243.44
Processing costs	799.5	8.34	178.68
General & administrative costs	291.4	3.04	65.13
Total cash operating cost	2,180.3	22.73	487.24
Net operating margin	3,082.9	32.15	688.94
Capital expenditure	444.3	4.63	99.30
Net cash flow (before tax)	2,638.5	27.51	589.64
Taxation	952.4	9.93	212.85
Net cash flow (after tax)	1,686.1	17.58	376.80

Annual cash flows are presented in Figure 1.7.

Figure 1.7
Life-of-Mine Cash Flows



1.16.4 Base Case Evaluation

In addition to the expansion capital of \$310.8M, the project has additional funding requirements for the acceleration of waste mining and increases in working capital, bringing the cumulative cash flow at the end of Year 2 to (negative) \$331.2M. From that point, with completion of the expansion project, the project demonstrates an undiscounted pay back of 3.7 years or approximately 4.4 years when the cash flow is discounted at the selected rate of 7.0%. The latter leaves a production tail of 8.0 years on the current mineral reserve.

Over the LOM period, the average cash operating cost equates to \$487/oz gold.

The base case evaluates to an IRR of 40% before tax and 30% after tax. At the selected discount rate of 7.0%, the net present value (NPV₇) of the cash flow is \$1,260.7M before tax and \$758.0M after tax.

The base case cash flow evaluation results are shown in Table 1.7, in which results at the comparative discount rates of 5%/y and 9%/y are also presented.

Table 1.7
Base Case Cash Flow Evaluation

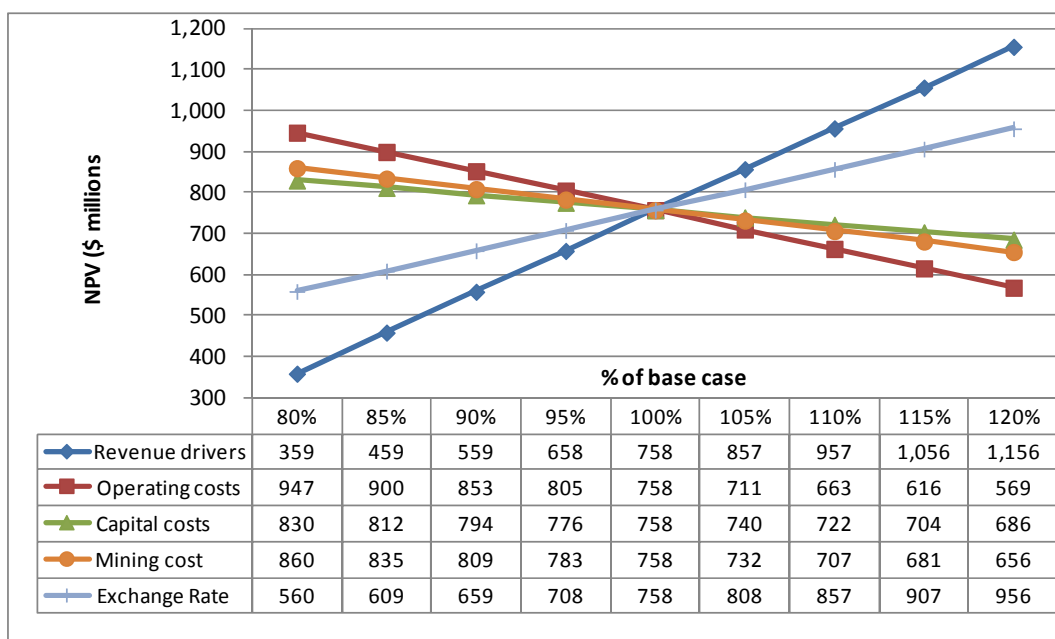
\$ million	LOM Total	Discounted at 5%/y	Discounted at 7%/y	Discounted at 9%/y	IRR (%)
Gross Sales	5,481.6	3,673.4	3,179.7	2,775.4	
Cost of Sales and Royalty	(218.4)	(146.5)	(126.8)	(110.7)	
Net Sales Revenue	5,263.2	3,526.9	3,052.8	2,664.6	
Mining Costs	1,089.3	841.3	767.1	703.2	
Processing Costs	799.5	542.4	471.6	413.5	
G&A costs	291.4	207.3	184.1	165.0	
Total cash operating costs	2,180.3	1,591.0	1,422.8	1,281.7	
Net Operating Margin	3,082.9	1,936.0	1,630.0	1,383.0	
Capital expenditure	444.3	387.2	369.3	353.6	
Net cash flow before tax	2,638.5	1,548.8	1,260.7	1,029.4	40%
Taxation payable	952.4	597.4	502.7	426.3	
Net cash flow after tax	1,686.1	951.4	758.0	603.1	30%

In Micon's opinion, the results demonstrate the economic viability of the project base case, under the conditions described above.

1.16.5 Sensitivity Study

The sensitivity of the project returns to changes in all revenue factors (including grades, recoveries, prices and exchange rate assumptions) together with capital and operating costs was tested over a range of 20% above and below base case values. Figure 1.8 shows the results of this analysis.

Figure 1.8
Sensitivity to Capital, Operating Costs and Revenue



In further analysis, Micon notes that simultaneously applying an increase of 57% to both capital and operating costs simultaneously would be required in order to reduce NPV₇ to zero. Likewise, a gold price of \$766/oz is required to produce a zero NPV₇, i.e., an economic break-even.

Micon concludes that the project is sufficiently robust to withstand adverse changes in the principal value drivers of the project, within the limits of accuracy of the estimate.

The sensitivity of project returns to changes in the gold price and discount rate were determined, as shown in Table 1.8.

Table 1.8
Sensitivity to Gold Price and Discount Rate

Gold Price		NPV (\$ million)		
\$/oz	Averages as of 31 Dec 2010	Discounted at 5.0%	Discounted at 7.0%	Discounted at 9.0%
1,000		528.9	392.1	283.5
1,023	36-month ave	572.3	429.7	316.4
1,100	24-month ave.	716.8	554.8	425.7
1,200		904.5	717.3	567.6
1,225	12-month ave.	951.4	758.0	603.1
1,300		1,092.2	879.8	709.5
1,391	1-month ave.	1,262.9	1,027.6	838.4
1,500		1,467.2	1,204.4	992.7
1,600		1,654.6	1,366.5	1,134.1
1,700		1,841.9	1,528.5	1,275.4
1,800		2,029.2	1,690.5	1,416.7
1,900		2,216.5	1,852.5	1,558.0
2,000		2,403.8	2,014.5	1,699.3

1.17 CONCLUSIONS

Micon concludes that this study demonstrates the viability of the project as proposed, and that expansion of the mine to process 20,000 t/d is warranted.

Prior to completion of the study, circumstances beyond the control of the study sponsor forced a suspension of work from December, 2010 until late in 2011. Nevertheless, it is Micon's opinion that the report remains current as of the signature date, despite the time elapsed since the effective date, which marks the completion of data collection for the study.

1.18 RECOMMENDATIONS

1.18.1 Geology

It is recommended that a drill program of approximately 50 diamond drill holes be located in strategic areas, within the VBK deposit to target areas where there is wide-spaced drilling

and Inferred Mineral Resource within the proposed VBK open pit boundary. Some of the proposed holes will be specifically designed to provide additional geotechnical data to confirm pit slope design. The holes would range between 240 and 686 m. The total depth of all holes would be approximately 25,000 m. All diamond drill holes should be collared HQ size (63.5 m) and if necessary reduced to NQ size (47.6 m). It is recommended that all Rusoro drilling and sampling procedures be followed to provide appropriate QA/QC controls and database validation.

The Coacia and Pisolita deposits have been drilled off to a depth of approximately 250 and 200 m below the surface, respectively. It is recommended that further drilling at the Coacia and Pisolita deposits be completed to target areas where there is wide-spaced drilling and Inferred Mineral Resources within their respective pit boundaries. A total of approximately 12,000 m in 60 diamond drill holes are recommended.

1.18.2 Geotechnical

Recommendations following the 2008/2009 geotechnical site investigation for the Tailings Storage Facility included:

- Collecting and testing additional “undisturbed” overburden samples
- Additional testing of in situ overburden
- Additional testing of bedrock samples, and
- Additional testing of select overburden and bedrock samples to confirm laboratory results.

It is further recommended that:

- Four untested Shelby tube samples be considered for triaxial testing.
- Subject to the above, collect additional Shelby tube samples for triaxial testing
- Block samples be collected for direct shear testing.
- Vane shear testing be incorporated into any additional geotechnical drilling to provide additional information regarding the overburden in situ shear strength parameters.
- Additional testing should be completed to confirm the UCS value of the bedrock gabbro and, if necessary, adjust the estimated UCS values.
- Additional laboratory testing of select overburden and bedrock samples should be conducted at a laboratory with well-known quality assurance and control standards to confirm the results from the GMK and UCAB laboratories. Select duplicate overburden samples were prepared for this purpose following the site investigation program and were being stored on site. The availability and condition of these samples should be confirmed.

- A geotechnical site investigation program should be carried out to confirm the parameters used in the open pit design, as this was not completed prior to this feasibility study.

1.18.3 Mining - Waste Storage

Waste rock dump seepage is monitored, but not collected. A seepage collection system is not planned for future dumps. It is recommended that the operation continues to monitor seepage and make modifications to future dump expansions and new waste rock dumps to allow for a collection and recovery system if seepage quality starts to degrade.

1.18.4 Metallurgy

Testwork has shown that, in general, 48-h CIL Au extractions of $\geq 90\%$ can be expected at $\sim 74 \mu\text{m}$, 40% solids, $\leq 0.5 \text{ g/L NaCN}$ and $\text{pH} \sim 11$, with typical consumptions of 0.6 kg/t NaCN and $<1 \text{ kg/t lime}$. Recommendations included:

- Careful metallurgical control due to head grade variability and different leach train processing required to optimize gold recovery.
- Tracking of CN-soluble gold for resource modeling purposes is recommended, as a basis for blending a more predictable feed.

1.18.5 Schedule

The implementation schedule is based on delivery of the SAG and ball mills 55 weeks from the date of placing the order, assumed to be 18 weeks after Board approval to proceed. The scheduled mill installation and commissioning period is 37 weeks. The mills are likely to be the critical item of equipment required to complete the project and should, therefore, be tendered and a recommendation to purchase issued as soon as practical after the commencement of sufficient engineering.

Other long-lead items such as transformers and the primary crusher will also need to be ordered near the commencement of engineering.

1.18.6 Power Supply

The source of permanent site power for the expansion has not been definitively resolved for this study, and the schedule impact of the power supply options is uncertain. Supply of permanent power for the expansion has the potential to become the schedule critical path.

1.18.7 Cost Estimates

The capital estimates presented in this section have been compiled by Micon from information provided by Ausenco and Knight Piésold during 2010. In Micon's opinion, the impact on these estimates of inflation and exchange rate variances between the effective date and the signature date of this report lie within the expected range of accuracy of the estimate.

However, due to the passage of time since the preparation of the capital and operating cost estimates used in this Feasibility Study, it is recommended that, subject to project approval, an update or revalidation of the estimates should be prepared as a basis for management of the implementation phase.

2.0 INTRODUCTION

Rusoro Mining Limited (Rusoro) operates two gold mines located in the vicinity of El Callao, Bolívar State, in eastern Venezuela: the Choco 10 / Choco 4 / Increible 6 licence area (the Choco area) and the Isidora underground mine. Output from both mines is processed through the Choco mill, which is owned and operated by Rusoro. The company has a 95% ownership interest in the Choco area, whereas the Isidora mine is subject to a joint venture with the Venezuelan government. Rusoro also has several early to advanced stage exploration projects in the El Dorado, Cuyuni and Km 88 districts of Bolivar State.

Within the Choco area, open pit mining currently takes place on the Rosika/Coatia/Pisolita deposits. Exploration has been carried out on the adjacent Villa Balazo-Karolina (VBK) deposit, as well as at the Elisa, Culebra and Christina deposits with the Increible 6 licence.

2.1 TERMS OF REFERENCE

Micon has previously been engaged to carry out a number of assignments for Rusoro:

In 2007, Micon was retained to provide an independent audit, compliant with requirements of National Instrument 43-101 (NI 43-101), of the Choco 10 mineral resource estimate and to report on the technical aspects of the Choco 10 property and operating mine. This request was made following the announcement that Rusoro had reached an agreement with Gold Fields Netherlands Services BV (Gold Fields), a wholly-owned subsidiary of Gold Fields, whereby Rusoro acquired all of Gold Fields' Venezuelan assets, including the producing Choco 10 mine. This report was filed on SEDAR on November 29, 2007 and is entitled "Technical Report on the PMG (Gold Fields) Choco 10 Concession and Mine, Estado Bolivar, Venezuela", dated November 21, 2007 (Leader et al., 2007).

Micon was further retained by Rusoro to carry out a preliminary assessment of the Choco 10 and Increible 6 properties. This report, referred to herein as Buchanan et al., 2009 was filed on SEDAR on June 9, 2009 and is entitled "Technical Report on the Preliminary Assessment of the Expansion of Production at Choco 10, Bolivar State, Venezuela", dated June 3, 2009 (Buchanan et al., 2009).

The mineral resource estimates documented in these reports were audited from the work undertaken by Gold Fields in 2006. Since that date, further definition and delineation drilling has been completed and Micon was retained by Rusoro in July, 2009 to undertake an update of the mineral resource estimates for the Choco area. The results of that exercise were presented in Micon's "Technical Report on the Mineral Resources of the Choco 10 Deposits, Bolivar State, Venezuela" dated August 18, 2010. The mineral resource estimates in that report utilized all drilling, assay and survey data up to December 31, 2009.

In late 2009, Rusoro further retained Micon to complete a feasibility study on the expansion of the Choco mill to 20,000 t/d, in line with the recommendations of a preliminary assessment completed earlier that year (Buchanan et al., 2009). This report presents the findings of that feasibility study.

2.1.1 Qualified Persons

The Qualified Persons responsible for the preparation of this report are as follows:

Name	Company	Area	Responsibility for Sections of this Report
David Makepeace, P.Eng	Micon	Mineral Resources	4-12, 14, 23, 26
Dayan Anderson, MMSA	Micon	Mining and production schedule, mineral reserves	15, 16, 21.2.1
Greg Lane, FAusIMM	Ausenco	Engineering design, process and infrastructure capital cost estimate	17, 18 (excl. 18.10)
Richard Gowans, P.Eng	Micon	Metallurgy, process and G&A operating cost estimates	13, 21 (excl. 21.2.1)
Daniel Friedman, P.Eng	Knight Piésold	Tailings Management	18.10
Christopher Jacobs, CEng MIMMM	Micon	Economic Analysis	1- 3, 19, 20, 22, 24 - 26

2.1.2 Site Visits

There have been several site visits by Micon and affiliated qualified persons since 2007:

Mr. David Makepeace, P.Eng., senior geologist with Micon visited the Choco 10 property and mining operations on July 29 to August 9, 2009.

Ms Jenifer Hill, R.P.Bio., senior environmental scientist with Micon visited the Choco facilities between August 14 and 16, 2007.

Mr. Daniel Friedman, P.Eng., senior engineer with Knight Piésold, visited the property from November 26 to December 4, 2008.

Mr. Robert Braun, B.E., MAusIMM, principal metallurgist with Ausenco, and Mr. Christopher Jacobs CEng MIMMM, senior mineral economist with Micon, visited the property and operation on October 19 to 24, 2009 together with other professional staff from each of their firms.

2.1.3 Study Preparation

Preparation of this report commenced with the site visits described above and continued with preparation of mineral resource estimates, metallurgical testwork, basic engineering and estimation of capital and operating costs through 2010.

However, prior to completion of the study, circumstances beyond the control of the study sponsor forced a suspension of work until late in 2011. Nevertheless, it is Micon's opinion that the report remains current as of the signature date, despite the time elapsed since the effective date, which marks the completion of data collection for the study.

2.2 UNITS AND CURRENCY

Quantities are generally stated in Système International d'Unités (SI) metric units, the standard Canadian and international practice, including metric tonnes (t), kilograms (kg) and grams (g) for weight, kilometre (km) or metres (m) for distance, hectares (ha) for area, and grams per tonne (g/t) for gold grades (g/t Au). Gold grades may also be reported in parts per million (ppm) or parts per billion (ppb). As is customary in the industry, quantities of gold may be reported in troy ounces (oz).

Table 2.1 provides a list of the abbreviations used in this report.

Table 2.1
List of Abbreviations

Term	Abbreviation
ALS Ammtec Limited	Ammtec
Ampere	A
Atomic Absorption Spectroscopy	AAS
Bolivar Fuerte (Venezuelan currency)	BsF or VEF
Bolivar Gold Corp	Bolivar
Capital Asset Pricing Model	CAPM
Carbon in Leach / Carbon in Pulp	CIL/CIP
Central Bank of Venezuela	CBV
Corporación Venezolana de Guayana C.A.	CVG
Coarse (crushed) Ore Stockpile	COS
Cubic metre(s)	m ³
Degree	°
Digital Terrain Model	DTM
Dollar(s)	\$ or US\$
Effective Grinding Length	EGL
Environmental Management System	EMS
Fibre Reinforced Plastic	FRP
General Mining de Guyana, C.A	General Mining
Gold Fields Limited	Gold Fields
Gram(s) / kilogram	g / kg
Grams per tonne	g/t
Hectare(s)	Ha
Hydrochloric Acid	HCl
Inflow Design Flood	IDF
Internal Rate of Return	IRR
Inverse Distance Cubed	ID ³
Lakefield Research Laboratories	Lakefield
Life of Mine	LOM
Litre	L
Maximum Design Earthquake	MDE
McClelland Laboratories, Inc.	McClelland
Metre(s) / centimetre / kilometre	m / cm / km
Metres above mean sea level	Masl

Term	Abbreviation
Micon International Limited	Micon
Micron	Mm
Mineral Resource Management	MRM
Million	M
Million tonnes	Mt
Million years old	Ma
Ministry of Basic Industry and Mines	MIBAM
Ministry of the People's Power for the Environment	MinAmb
Motor Control Centre	MCC
Net Present Value (discount rate)	NPV _(x)
Not applicable	n.a.
Ounce(s) [troy ounce]	oz
Parts per million	ppm
Parts per billion	ppb
Probable Maximum Flood	PMF
Process Research Associates Ltd.	PRA
Promotora Minera de Venezuela SA	Promiven
Promotora Minera de Guayana S.A.	PMG
Rosika/Coacia/Pisolita	RCP
Rock Quality Designation	RQD
Run of Mine	ROM
Rusoro Mining Ltd	Rusoro
Semi Autogenous / Ball Mill /Crusher (flowsheet)	SABC
Semi Autogenous Grinding	SAG
SAG Mill Comminution	SMC
Sodium Cyanide	NaCN
Sodium Hydroxide	NaOH
Square metre(s)	m ²
Square kilometre(s)	km ²
Summit Valley Equipment & Engineering	SVEE
System for Electronic Document Analysis and Retrieval -	SEDAR
Système International d'Unités	SI
Tailings Storage Facility	TSF
Tonne(s)	t
Tonnes per day	t/d
Uniaxial Compressive Strength	UCS
Villa Balazo-Karolina	VBK
Volt / kilovolt / Megavolt	V / kV / MV
Waste Water Treatment Plant	WWTP
Watts / kilowatts / megawatts	W / kW / MW
Weighted Average Cost of Capital	WACC

3.0 RELIANCE ON OTHER EXPERTS

Micon has reviewed and analyzed data provided by Rusoro and others, and has drawn its own conclusions, augmented by its direct field examinations. Micon has not carried out any independent exploration work, drilled any holes or carried out any extensive sampling and assaying on the Choco 10 or Incredible 6 properties. During the field visit to Choco 10 and Incredible 6, Micon did not collect any samples to confirm the mineralization as it is an operating gold mine and any samples collected by Micon would only reflect the mineralization at the sample location and not necessarily the economic nature of the mineralization at the mine.

While exercising all reasonable diligence in checking, confirming and testing, Micon has relied upon Rusoro's presentation of the Choco 10 and Incredible 6 data from both itself and previous organizations in formulating its opinion.

Micon has not reviewed any of the documents or agreements under which Rusoro holds title to the Choco gold mine or its underlying mineral concessions and Micon offers no legal opinion as to the validity of the mineral titles claimed. A description of the properties, and ownership thereof, is provided for general information purposes only. Rusoro has confirmed the description presented in Section 4 of this report.

The existing environmental conditions, liabilities and remediation have been described where required by NI 43-101 regulations. These statements also are provided for information purposes only and Micon offers no opinion in this regard.

The descriptions of geology, mineralization and exploration are taken from reports prepared by Rusoro, its predecessors or its consultants. The conclusions of this report rely on data available in published and unpublished reports and information supplied by the organizations which have conducted exploration on the property, and information supplied by Rusoro and its consultants. In Micon's opinion, the information provided to Rusoro was supplied by reputable organizations and Micon has no reason to doubt its validity.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Choco 10 gold mine is located in the eastern part of Venezuela, in Bolívar State (see Figure 4.1). It is approximately 15 km west of the town of El Callao. The location of the mine is approximately 7° 19' 27" N and 61° 52' 20" W or UTM coordinates 809740.7 m N, 624478.4 m E, Zone 20.

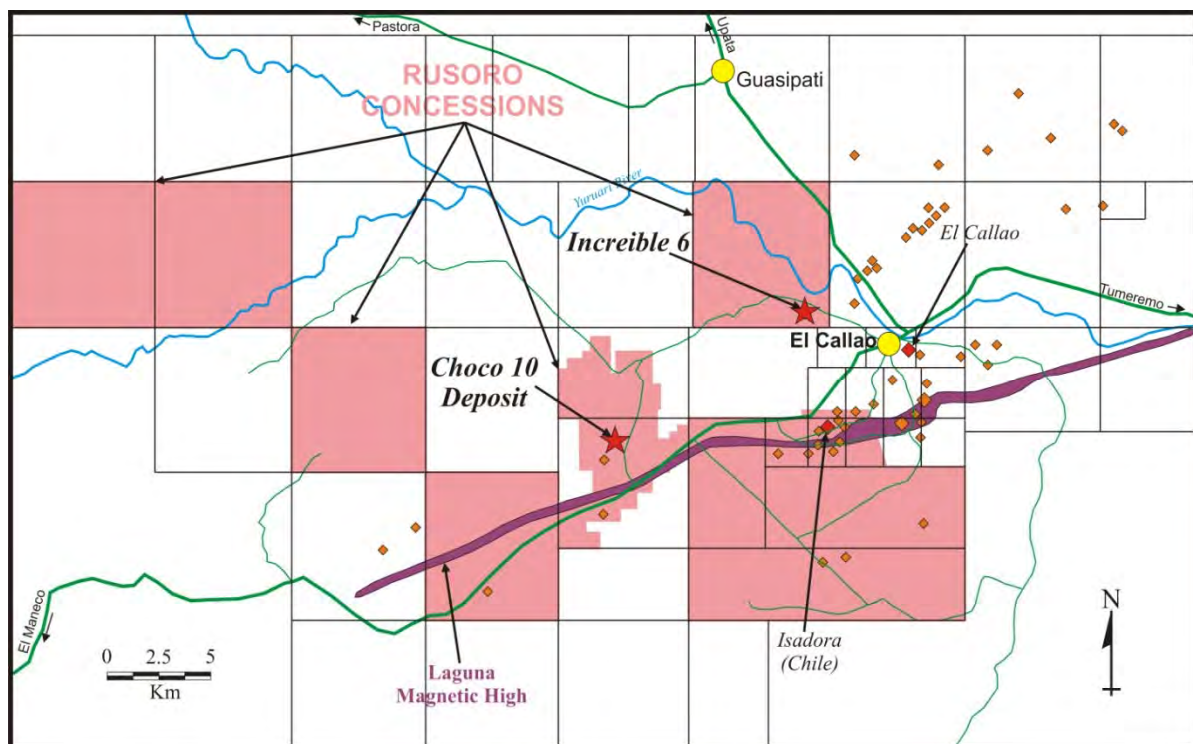
Figure 4.1
Location of the Choco 10 Mine, Venezuela



The major industrial city of Puerto Ordaz (Ciudad Guayana) is located 190 km northwest of El Callao and is linked to the mine by paved roads. Although Venezuela has a good road infrastructure, road conditions near the mine have deteriorated during the last 15 years. Under the terms of its exploitation certificate the Company is obligated to maintain a portion of the access road from El Callao to the Choco 10 mine.

The mine and mill are located on an exploitation permit, which amalgamates the Choco 10 and Choco 4 concessions (Figure 4.2).

Figure 4.2
El Callao District Showing Locations of Rusoro Concessions



The Choco 10 mine commenced production in August, 2005 utilizing typical drill, blast, load and haul open pit mining and processing of ore through a conventional comminution and carbon-in-pulp (CIP) plant. The Rosika/Coacia/Pisolita (RCP) combined open pit currently provides all ore processed from within the Choco 10 concession. The pit is located 2-3 km from the mill site.

4.2 CLAIMS

4.2.1 Choco Concession

The Choco 10 Mine property consists of two mineral concessions (Choco 4 and Choco 10). However, the known mineral resource lies entirely within the Choco 10 concession.

The mining rights over the original area (4,249.35 ha) were first granted by the Ministry of Energy and Mines (MEM) to Corporación Venezolana de Guayana C.A. (CVG) by means of a concession on May 10, 1993 (Official Gazette No. 4.578 Extraordinary, May 18, 1993). It was registered before the Real Estate Registry Office of the Roscio District, Bolívar State, on April 15, 1993. The concession was leased by CVG to Promotora Minería de Guayana (PMG). A Certificate of Exploitation was granted on February 4, 1994 and PMG selected an exploitation area of 2,124.53 ha divided into 5 plots. It was authenticated before the Second Notary Public of Puerto Ordaz, Bolívar State, on April 8, 1994, under No. 7 and 9, Volume 55 of the Authentication Books of the Notary and registered before the Real Estate Registry

Office of the Roscio District, Bolívar State, on April 15, 1993 as Choco 4 and 10 Lease Agreement. The agreement grants PMG the right to explore and exploit alluvial and vein material (manganese, niobium, tantalum, molybdenum, vanadium, chromium, nickel, cobalt, tungsten, gold, copper, zinc, silver and tin), subject to the payment of royalties, taxes and meeting a number of obligations. Gazetting of the final approval for issuance of exploitation for Choco 10 was completed on December 5, 2005 with a term of 20 years which terminates December 5, 2025.

4.2.2 Increible 6 Concession

The Increible 6 concession is 100 % owned by General Mining de Guayana C.A., a wholly owned company of Rusoro. It has both hard rock and alluvial rights. It is 2,470.53 ha in size and can be seen in Figure 4.2. The Certificate of Exploitation for the Increible 6 concession was approved in December, 2008. The Ministry of the Peoples's Power for the Environment (MinAmb) issued the authorization to affect Natural Resources for Increible 6, in late 2010. This was the final step in the permitting process toward the commencement of mining activities at Increible 6.

4.3 ROYALTIES

Bolivar Gold Corporation (Bolivar) acquired Promotora Minería de Venezuela (Promiven) and its 70% interest in PMG in 2003. Bolivar agreed to pay the previous owners of PMG, Cemex Venezuela C.A. (Cemex), a royalty on gold production once it reaches 700,000 oz. The royalty is calculated monthly based on the average London PM fixed price of gold during the calendar month. The minimum royalty is US\$ 10 per oz. If the price of gold exceeds US\$ 315 per oz in the relevant calendar month the royalty is US\$ 15 per oz. If the price of gold exceeds US\$ 400 per oz in the relevant calendar month the royalty is US\$ 20 per oz. Bolivar has the right of first refusal to purchase Cemex's right to receive the royalty.

In addition, a monthly production royalty must be paid to CVG and its subsidiary, CVG Técnica Minería C.A., with respect to the lease agreement mentioned in Section 4.2.1. This royalty is calculated monthly based on gold production and ranges between 1.0 to 3.5% with respect to the average price of gold in the New York market for the relevant month as determined by CVG. This royalty is subject to value added tax.

4.4 PERMITS AND LICENSES

Based on discussions with Choco staff, official gazettes, exploitation permits and monthly and quarterly reports, Micon understands that all permits and licenses are in place to operate the mine. In addition to the rights and titles discussed previously, an authorization was granted to occupy Choco 10 (No. 000580) which expires on February 4, 2014. The environmental permits are in place for mineral exploitation in the both concessions. The mine is currently fulfilling the permit requirements. A groundwater well field permit has been obtained and is being complied with. Permits for open pit expansion, waste rock dump sites, mill expansion and new tailings pond impoundment facilities are in progress.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following has been extracted from Leader et al., 2007.

5.1 ACCESS

A paved secondary road passes through the concessions near the Choco 10 mine and mill. This road provides access to the town of El Callao, about 15 km to the east from the concessions (see Figure 5.1). El Callao has a population of approximately 25,000 and is the centre of population in the area. It is a historic gold mining centre and a number of present and past producing mines are located nearby. The main highway in the region passes on the outskirts of El Callao. Puerto Ordaz, a major city on the Orinoco River, lies 190 km to the north. The main highway also connects other gold mining centres of El Dorado and Kilometre 88 further south. The highway continues to the Brazilian frontier.

The Choco concessions are mostly unused for agricultural activity and the gently sloping topography provides suitable and adequate locations for project infrastructure, waste rock storage and tailings management facilities. Figure 5.1 is a general plan showing the major mining and infrastructure areas within the concessions.

5.2 CLIMATE AND PHYSIOGRAPHY

Choco 10 and Incredible 6 concessions lie in an area of low hills between elevations 200 masl and 300 masl.

The climate at Choco is tropical, with temperatures averaging around 25.7°C and humidity ranging from 76% to 82%. The estimated mean annual precipitation at El Callao is 1,325 mm. The heaviest rainfall typically occurs during the months from June to August, which averages 150 mm of precipitation per month. During the remainder of the year the rainfall is about 80 mm per month except in the dry season from February through March when precipitation is about 36 mm per month.

The Yuruarí River is the major watercourse in the area which runs through El Callao. It is several km to the north and east of the mine site.

5.3 INFRASTRUCTURE

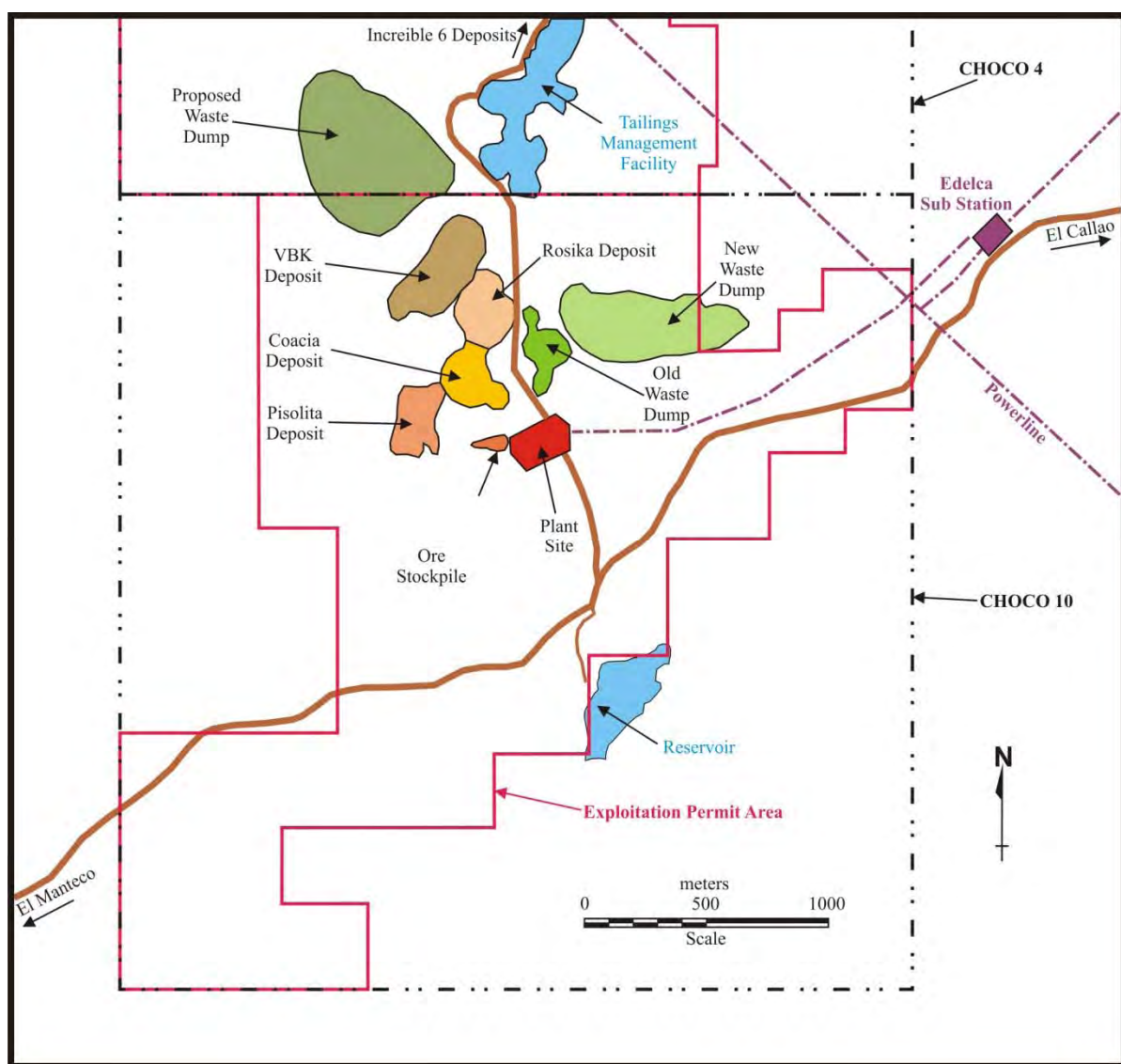
A major high voltage 400 kV electrical supply line carrying power from the Guri Dam on the Caroni River, southwest of Puerto Ordaz, and destined for Brazil, passes between the mine site and El Callao. A substation immediately east of the Choco concessions supplies power to El Callao and the mine.

A rain-dependent reservoir supplies water for use at the mine, which is supplemented by a groundwater well field. A potential water supply is from the Yuruarí River, which passes

north of the concessions, approximately 8 km distant, and from which other local mines and the El Callao municipality draw water.

Most of the personnel working at the mine live in and around El Callao. The population is familiar with mining and can provide the majority of the local labour force. Mine supervision and technical support are provided by experienced Venezuelan management and contract expatriate staff that are housed in staff quarters on the outskirts of El Callao and Tumeremo. Personnel travel by bus or truck to the mine each day and there is no permanent accommodation at the mine site.

Figure 5.1
Plan Showing the Choco 10 Mining Areas and Mine Infrastructure



6.0 HISTORY

For details on the history of the Choco properties, the reader is referred to Leader et al., 2007 and Laudrum et al., 2008. The following has been extracted from those reports.

Gold mining in the El Callao district dates back to the early Spanish Conquest in the 1500s in the search to find El Dorado. The centre of the Spanish colony was believed to be Tupequen, modern day El Callao.

During the 1800s, exploitation of gold in the El Callao district was undertaken first by a Frenchman, Dr. Luis Plassard, in 1849 and then by Corsicans and Venezuelans who established the El Callao Company, in 1870. The company mined the El Callao mine below the site of the present town until 1887.

6.1 CHOCO 10

Mining in the area of the Choco 10 concessions date back to 1897, when a British company operated the historic Concordia mine located 2 km from the present Choco 10 operation. A mine shaft was sunk 200 m to develop a 1 m wide quartz vein reportedly carrying 40 g/t of gold.

New Goldfields of Venezuela (a subsidiary of Gold Fields) commenced mining in the El Callao district in the second part of the 1920s and acquired a number of properties including Choco 6 in 1921 and Choco 1, 2, 3, 4, and 5 in 1924. Modern exploration activity in the Choco area started in the early 1990s with a regional geochemical soil survey carried out within the Choco 4 and 10 concessions.

On October 11, 2007, Rusoro acquired all of the Venezuelan assets of Gold Fields, including the Choco 10 gold mine. Prior to Rusoro's acquisition, Gold Fields held the rights to Choco 10 and other Venezuelan assets that were acquired through a plan of arrangement with Bolivar on February 28, 2006. Under the plan of arrangement Gold Fields acquired all of the outstanding securities in Bolivar which it did not already hold. Gold Fields owned its interest in the Choco 10 mine through its holding in PMG which remains the operating company for Choco 10. PMG is owned 5% by CVG and 95% by Promiven, the latter being wholly-owned by Rusoro through Venezuela Holdings (BVI) Ltd. and Carisma Corporation AVV.

6.2 INCREIBLE 6

There has in the past been a significant amount of shallow artisanal gold mining within the Increible 6 concession. There are no records of when this work commenced or the total production to date. Essentially, all of the areas explored by Rusoro have previously been explored and mined on a small scale. All of this previous work is confined to the upper, weathered zone.

In 1989, General Mining was granted a one-year licence to prospect on the Incredible 6 concession. This licence was subsequently extended for a second year. During that time, favourable results from the sampling of veins in three areas were obtained.

In the late 1980s, CVG Minera (CVG), a federally chartered regional development corporation, was formed for the purpose of promoting development in southeastern Venezuela. It was granted the Incredible 6 concession on January 23, 1991. CVG then transferred the concession to General Mining under a two-year contract, dated May 13, 1991, covering alluvial and vein deposits. The contract was extended for an additional year and a three year exploration program was commenced. The size of the concession was reduced by 50% after the three years.

Through a wholly-owned subsidiary, Rusoro acquired the property in 2004 and has been actively exploring the Incredible 6 property since that time.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The following descriptions of the regional and local geology pertaining to the Choco 10 property is based on previous Micon reports (Leader et al., 2007 and Buchanan et al., 2009).

7.1 REGIONAL GEOLOGY

The Choco mineral concessions are located within the Guyana Shield, which occupy the northern part of the Amazon Craton between the Amazon and Orinoco river basins. The geology of the Guyana Shield as a whole is poorly known, reflecting minimal development, limited access, as well as intense tropical weathering and cover across this large area. It has been subdivided into three major geological entities:

- Archaean rocks of the Imataca Complex.
- Palaeoproterozoic Trans-Amazonian granite-greenstone belts.
- Palaeoproterozoic sedimentary and igneous rocks of the Roraima Group, Uatumã Group and the Avanavero Suite.

The majority of gold mineralization is hosted by the greenstone belts; some of the more important of these include the Pastora Supergroup and Botanamo Group in Venezuela, the contiguous Barama-Mazaruni Groups in Guyana, the Marowijne Group in Suriname, and the Maroni Group in French Guiana (Figure 7.1). Greenstone belts across the shield are dated to between 2,250 and 2,110 Ma for the metavolcano-sedimentary sequences and between 2,250 and 1,900 Ma for the associated granitoid complexes. The volcano-sedimentary packages and early granitoids were metamorphosed, deformed and mineralized during the Trans-Amazonian Orogeny, dated from approximately 2,200 to 1,900 Ma. This orogeny caused the accretion of various volcanic centres (the precursors of greenstone belts) around Archaean palaeocontinents including the Imataca Complex in Venezuela.

The Palaeoproterozoic greenstone belts and granitoid complexes are separated from the Imataca Complex by the Guri Structure, a major eastnortheast- trending ductile shear zone (see Figure 7.2). At a regional scale, many gold deposits of the Guyana Shield are located in close proximity to, but not on, major shear zones as is typically the case for orogenic gold deposits.

Venezuela has a long history of gold production. The most important area was the El Callao mining district, which hosts the Choco 10 and Incredible 6 deposits. The El Callao Mine is considered to have been the most productive gold mine in the world during the latter part of the 19th century. As of 1995, more than 260 gold-bearing quartz veins were known to exist in the district (MDA, 2005).

Figure 7.1
Regional Geology of the Northern Guyana Shield Showing Major Gold Deposits

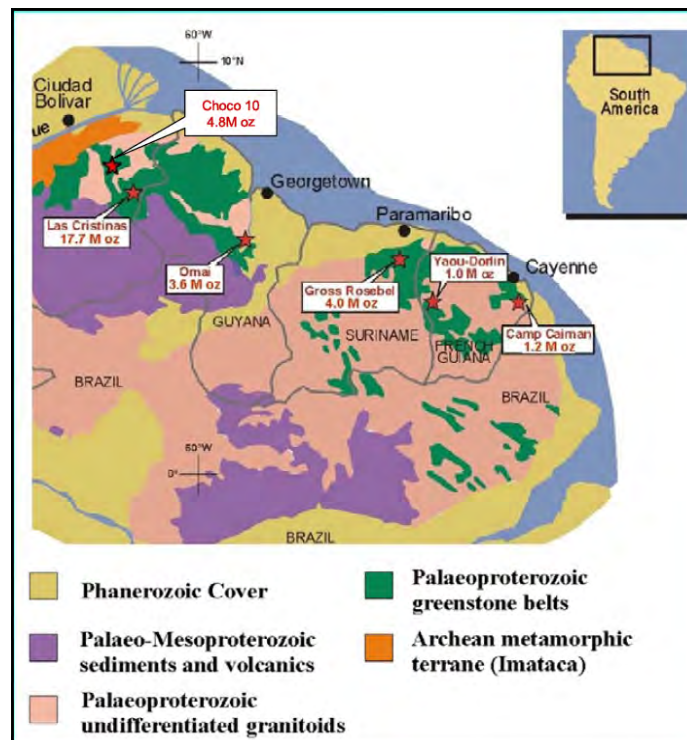
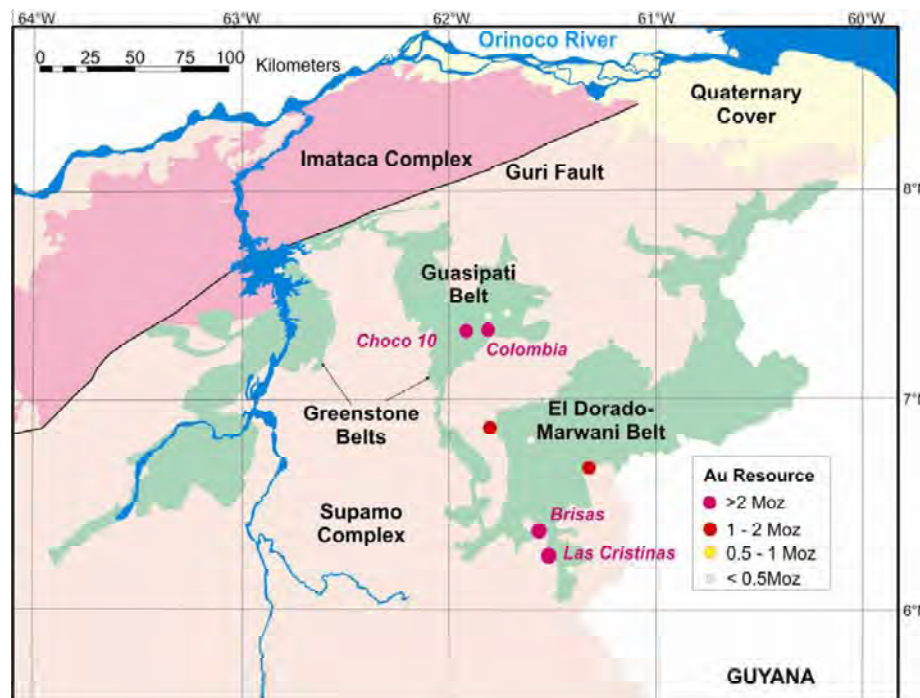


Figure 7.2
Regional Geology of the Choco 10 and Other Major Gold Deposits



7.2 LOCAL GEOLOGY

The gold mineralization at Choco 10 and Incredible 6 deposits are typical of Archaean-Proterozoic orogenic gold deposits. The deposits are hosted in an early Proterozoic sequence of the Guasipati Greenstone Belt (or Pastora Supergroup) of the Guyana Shield. The stratigraphy comprises a tholeiitic to calc-alkaline volcanic package, overlain by volcanoclastic and epiclastic rocks intruded by gabbroic sills. The rock package has been subjected to intense tropical weathering. The structural architecture is dominated by folds and ductile fabrics indicating a long history of compressional deformation. Mineralization is dominantly structurally controlled and is associated with strain partitioning. High-grade gold mineralization occurs with carbonate, pyrite, silicification and quartz-veining in lower-strain zones typically associated with crenulations, folding and chaotic foliations.

The deposits have been metamorphosed in the lower greenschist facies. Late dykes of intermediate composition and felsic stocks and sills have also intruded the greenstone-belt package.

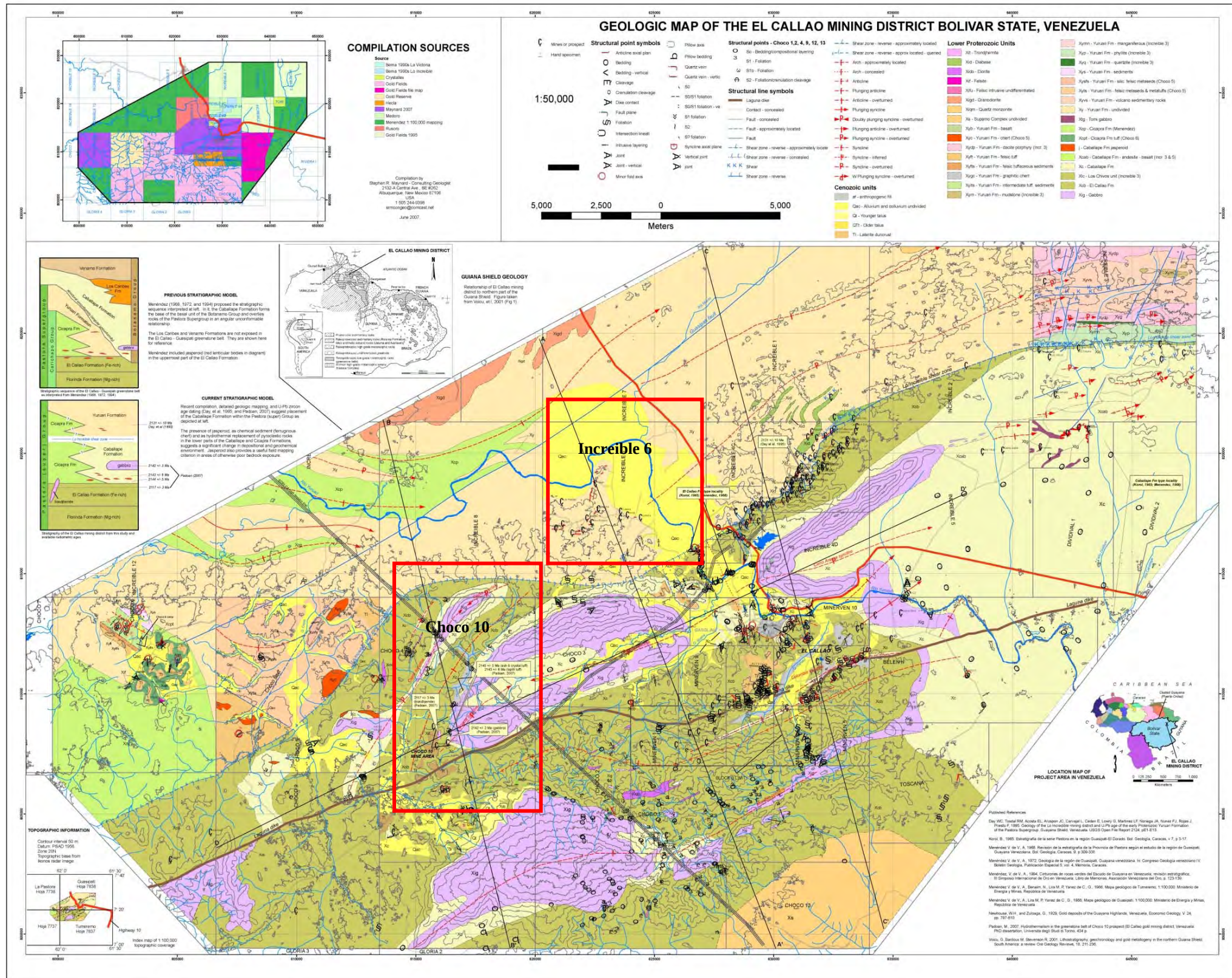
The meta-volcanics and meta-sedimentary rocks that underlie this area can be summarized, in ascending stratigraphic order, as:

- **El Callao Formation:** tholeiitic basalts; mainly pillowed with hyaloclastite breccias and massive zones.
- **Cicapra Formation:** tholeiitic basaltic package; volcanoclastic rocks, with conglomerates and breccias at the top of the sequence, brecciated basalt flows, some arkosic sandstones in the core of the formation.
- **Caballape Formation:** intermediate calc-alkaline volcanoclastic rocks, breccias, conglomerates, tuffs, siltstones, epiclastic–pyroclastic.

The granitic domes of the Supamo Complex divide packages of meta-sedimentary and meta-igneous greenstone-belt rocks to form branching synclinoria between the intrusive uplifts. Structural geology of the greenstone belts is dominated by folds and ductile fabrics indicating a long history of compressional deformation. The Choco mine area is located immediately north of the Laguna Dyke which cuts north-northeasterly through the area. All of the currently known major gold deposits in the district are reported to occur to the north of this dyke. The distribution of the stratigraphy, together with the more prominent structural geological features, is illustrated in Figure 7.3.

The rock package has been subjected to intense tropical weathering. Consequently, much of the near-surface mineralization is contained within saprolitic horizons.

Figure 7.3
Local Geology of the El Callao District



7.3 LOCAL DEPOSITS AND MINERAL OCCURRENCES

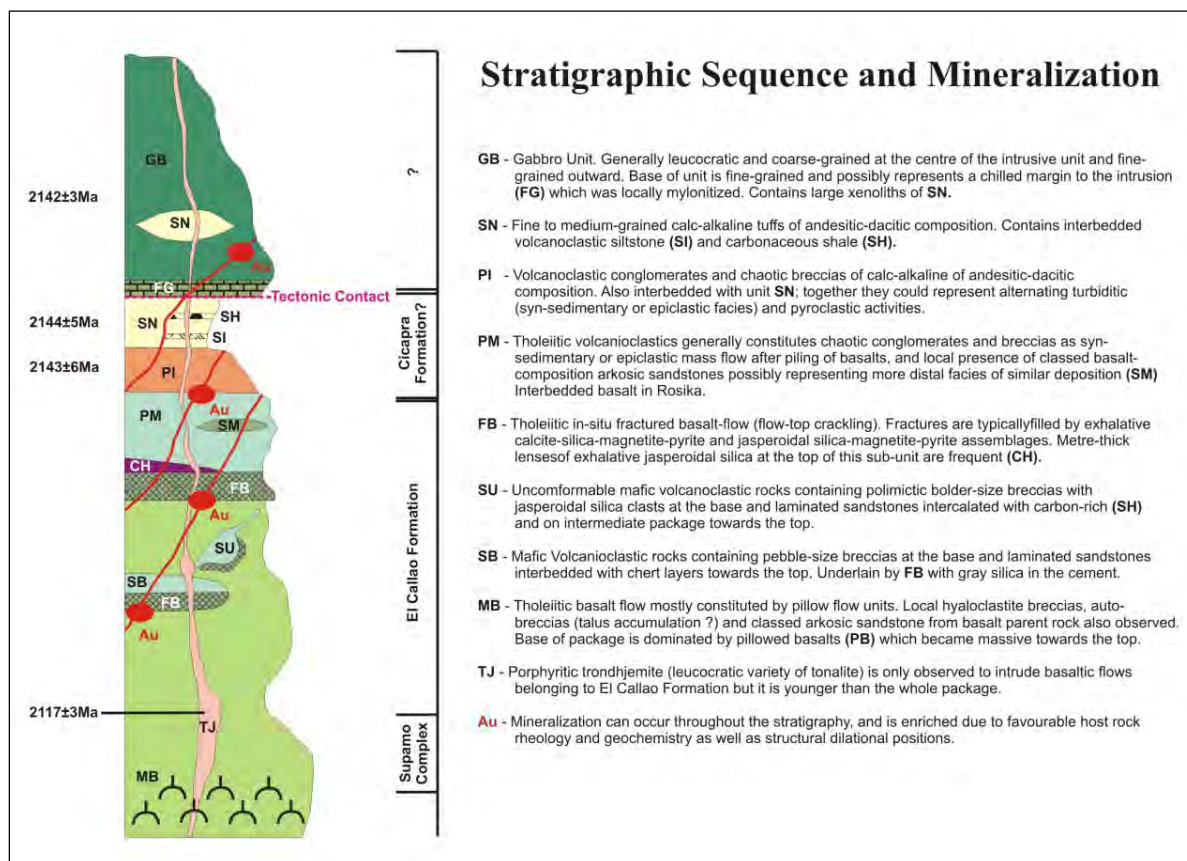
7.3.1 Choco 10 Deposits Geology

The Choco 10 gold complex is located in a regional northeast-plunging synclinal hinge zone. Mineralization is dominantly structurally controlled and is associated with strain partitioning. High-grade gold mineralization occurs with carbonate, pyrite, silicification and quartz-veining in lower-strain zones typically associated with crenulations, folding and chaotic foliations.

The current stratigraphic sequence established for the Choco 10 deposits is shown in Figure 7.4. From stratigraphic base to top (west to east), the main rock units in the camp are:

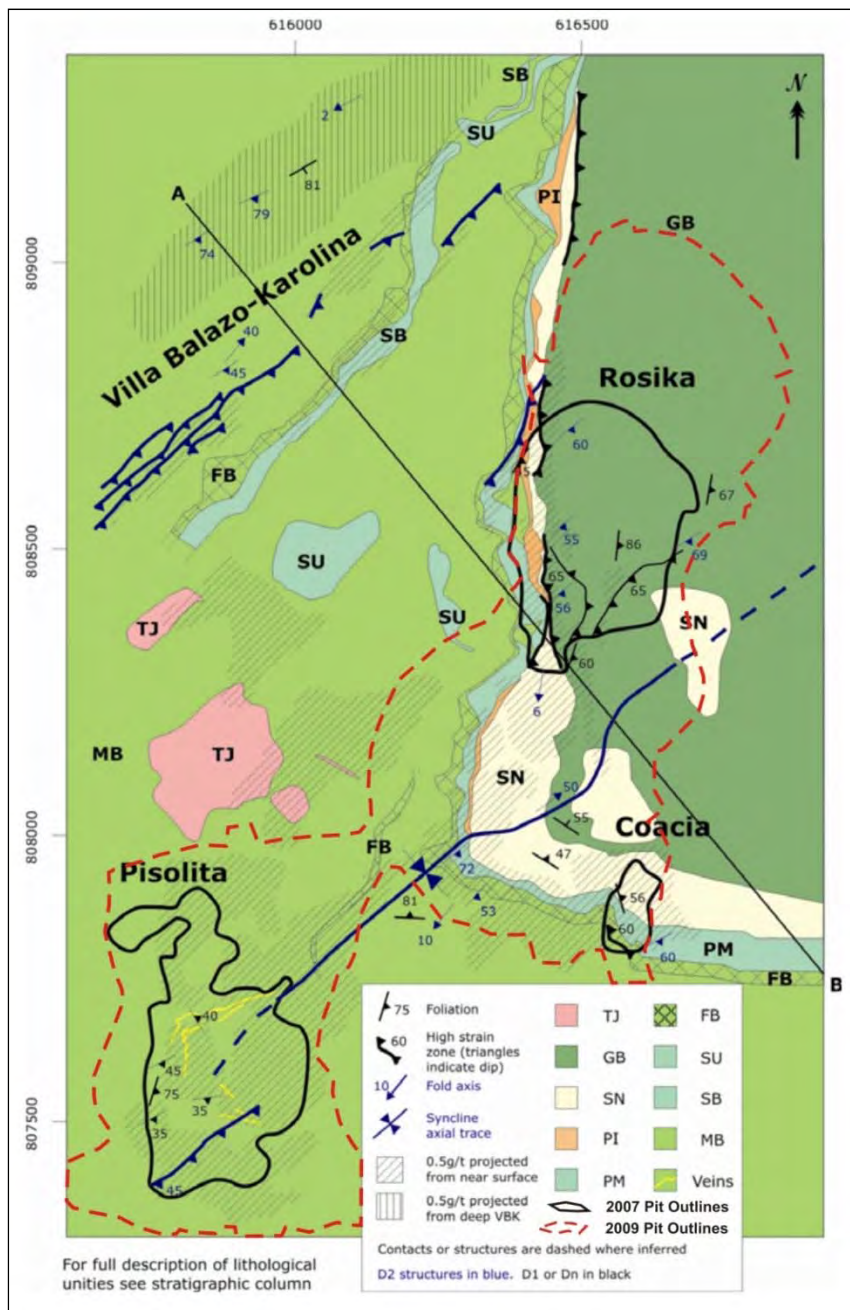
- Tholeiitic basalts, flow-top breccias and interbedded chert.
- Mafic volcanoclastic rocks.
- Calc-alkaline intermediate volcanoclastic sediments.
- Gabbro sill intruding both the basalts and volcanoclastic units.
- Trondhjemite pluton and associated dykes intruding the whole sequence.

Figure 7.4
Stratigraphic Sequence and Mineralization of the Choco 10 Deposit



The deposits are located in a regional northeast-plunging synclinal hinge zone (Figure 7.5). The structural architecture is dominated by folds and ductile fabrics indicating a long history of compressional deformation. Deformation partitioning is highly developed and, as a result, there are large volumes of rock that preserve primary features (low-strain zones), while deformation is concentrated in zones of intense development of ductile fabrics (high-strain zones). Ductile or brittle shear zones that accommodate large displacements are absent.

Figure 7.5
Local Geology, Choco 10



The characteristics of gold mineralization at the Choco 10 deposits are varied as a result of a complex interplay between host rock rheology, permeability and geochemistry, proximity to structures and hydrothermal fluid composition. Higher-grade mineralization is dominantly hosted in lower strain domains characterized by spaced or chaotic crenulation and less commonly brecciation and cataclastites, coupled with well-developed continuous foliations typical of high-strain domains.

In general, mineralization is accompanied by carbonate alteration, dominantly ankerite and dolomite, and varying degrees of silica and sericite. Pyrite is the most abundant sulphide phase; it occurs in relatively low concentrations (typically 2-10%) and has a good correlation with gold tenor.

Replacement-style hydrothermal alteration is common in more permeable host-rocks such as coarse-grained volcanics and flow-top breccias of basalts. In less permeable massive basalts and gabbros, vein-hosted mineralization is common and often contains visible gold. Magnetite, present as disseminations and stringers between clasts in the original flow-top breccias, has been replaced by pyrite which is closely associated with gold mineralization in the higher-grade lodes.

The four principal deposits at Choco 10 are Rosika (R), Coacia (C), Pisolita (P) and Villa Balazo-Karolina (VBK) (see Figure 7.5). The present open pit encompasses the Rosika, Coacia and Pisolita deposits and is referred to as RCP.

7.3.2 Rosika, Coacia and Pisolita

Within the continuous Rosika-Coacia system, rheological heterogeneities resulting from the distinct stratigraphic sequence form the dominant geometrical control on mineralization. Lithological contacts dip between 40° and 65° to the east, in the northern limb of the syncline. Towards Coacia, the stratigraphy steepens and wraps around the syncline. Mineralization broadly, but not strictly, follows this geometry. The most favourable sites for mineralization are the coarse intermediate volcanoclastic, the contact between the intermediate and mafic volcanoclastic units, and the flow-top breccia unit at the top of the footwall basalt. However, the entire stratigraphy (from hanging-wall gabbro, through volcanoclastics, to footwall basalt) can host economic mineralization.

The Pisolita deposit lies within the basal basaltic unit, with mineralization predominantly within the regolith. Mineralization is dominantly controlled by a stack of sub-horizontal quartz veins which formed in the southwest continuation of the hinge zone of the Coacia syncline.

7.3.3 Villa Balazo-Karolina

Mineralization at Villa Balazo-Karolina (VBK) is hosted within southeast dipping lower-strain mineral zones bounded by high-strain lower grade halos. The mineralized strain zones in VBK form a stacked series, largely due to the prevailing basalt-dominated stratigraphy which lacks such a strong fluid focus compared to the stratigraphic sequence hosting

Rosika-Coacia. Where favourable stratigraphy provides geochemical and rheological contrasts, mineralization is enhanced at the intersection with the strain zones, particularly in the flow-top breccias.

7.3.4 Incredible 6

The geology at Incredible 6 is dominated by volcanoclastic rocks, similar to the Choco 10 area. The concession is underlain by, from oldest to youngest, metabasalt and andesite, carbonaceous siltstones with conglomerate, felsic volcanics, greywackes and siltstones, and quartz-porphyry. The metavolcanic (tuff) rocks are the most extensive.

A gabbro-diorite intrusion is noted south of the property, and intrusive quartz porphyry (possibly sub-volcanic) is observed in various portions of the central part of the property. Exposures are limited on the property and there is typically 0.5 m to 2 m of overburden present. Nearer to the Yuruarí River, the overburden increases to greater than 4 m.

Deep weathering is present over most of Bolívar State and is typically up to 50 m thick in the Incredible 6 area.

A series of sub-parallel shears within the volcanics host the gold mineralization. Shears of this phase also cut the quartz porphyry. Low-sulphide quartz-carbonate veins that are syn-D2 (second phase deformation) carry the most significant economic gold concentrations. The alteration assemblage consists of chlorite, sericite, quartz, carbonate, and pyrite.

The geology in the area of Incredible 6 concession is shown in Figure 7.3.

7.4 MINERALIZATION

The following descriptions is a summary of the extract from Leader et al., 2007 and Laudrum et al., 2008, to which the reader is referred for more details.

7.4.1 Choco 10

The characteristics of gold mineralization of the Choco 10 deposits are varied as a result of a complex interplay between host rock rheology, permeability, geochemistry, proximity to structures and hydrothermal fluid composition. Mineralization is dominantly structurally controlled and is associated with strain partitioning. Higher-grade mineralization is dominantly hosted in lower strain zones characterized by spaced or chaotic crenulation and less commonly brecciation and cataclastites, coupled with well-developed continuous foliations typical of high-strain domains.

In general, mineralization is accompanied by carbonate alteration, dominantly ankerite and dolomite, and varying degrees of silica and sericite. Pyrite is the most abundant sulphide phase. It occurs in relatively low concentrations (typically 2-10%) and has a good correlation with gold grade.

Replacement-style hydrothermal alteration is common in more permeable host rocks such as coarse-grained volcanoclastics and flow-top breccias of basalts. In less permeable massive basalts and gabbros, vein-hosted mineralization is common and often contains visible gold. Magnetite, present as disseminations and stringers between clasts in the original flow-top breccias, has been replaced by pyrite which is closely associated with gold mineralization in the higher-grade lodes. Specific characteristics of the mineralization at each deposit are outlined below.

Within the continuous Rosika-Coacia system, rheological heterogeneities resulting from the distinct stratigraphic sequence form the dominant geometrical control on mineralization. The most favourable sites for mineralization are the coarse intermediate volcanoclastics, the contact between the intermediate and mafic volcanoclastics units, and the flow-top breccia unit at the top of the footwall basalt. However, the entire stratigraphy (from hanging wall gabbro, through volcanoclastics, to footwall basalt) can host economic mineralization.

Pisolita lies within the basal basaltic unit, with mineralization predominantly within the regolith. Mineralization is dominantly controlled by a stack of sub-horizontal quartz veins which formed in the southwest continuation of the hinge zone of the Coacia syncline. Mineralized zones in the fresh rock are considered to be analogous to VBK-style lodes. Pisolita is named for the significant mineralization hosted within a pisolitic laterite horizon.

Mineralization at VBK is hosted within southeast dipping lower-strain zones bounded by high-strain lower-grade halos. The mineralized strain zones in VBK form a stacked series, largely due to the prevailing basalt-dominated stratigraphy which lacks such a strong fluid focus compared to the stratigraphic sequence hosting Rosika-Coacia. Where favourable stratigraphy provides geochemical and rheological contrasts, mineralization is enhanced at the intersection with the strain zones, particularly in the flow-top breccias.

7.4.2 Increible 6

The Increible 6 concession is underlain by, from oldest to youngest, metabasalt and andesite, carbonaceous siltstones with conglomerate, felsic volcanics, greywackes and siltstones, and quartz-porphyry. Gold mineralization is hosted by a series of sub-parallel shears within the volcanics. Low-sulphide quartz-carbonate veins that are synchronous with second phase deformation carry the most significant economic gold concentrations. The alteration assemblage consists of chlorite, sericite, quartz, carbonate and pyrite.

Currently identified areas of significant gold mineralization on the Increible 6 concession are restricted to a deformation zone through the central part of the property. The area hosts numerous, discontinuous low sulphide, sericitic shear zones. Better grade gold values are typically related to shear-hosted quartz (+carbonate) veins and quartz porphyry bodies. Gold is present in both volcanic and intrusive rocks.

Within the deformation zone, a number of separate mineralized domains have been recognized. These domains consist of east-west striking sub-parallel zones with a moderate

southerly dip. From east to west, these zones of mineralization have been named Culebra, Cristina, Ingrid-Elisa, and Olga-Enoc.

There is a north trending zone, north of the Christina deposit that includes several small mineralized domains including the Payara zone. These small zones are not well defined and are not included in the resource estimate.

7.5 ALTERATION

On a regional basis, alteration is typified by a metamorphic assemblage that consists of epidote-chlorite with $\pm$ amphibole $\pm$ albite $\pm$ magnetite $\pm$ sericite $\pm$ carbonate. Low-grade halos adjacent to mineralization typically have alteration assemblages consisting of chlorite $\pm$ carbonate $\pm$ sericite $\pm$ magnetite and contain in the order of 100-500 ppb gold.

8.0 DEPOSIT TYPES

The following has been extracted from Leader et al., 2007, to which the reader is referred for more details on deposits in the El Callao district.

The Choco 10 and Incredible 6 deposits, together with other gold deposits in the El Callao district, are considered to be “orogenic-type” deposits. These types of deposits are sometimes also, very broadly, referred to as mesothermal. They are characteristically associated with deformed and metamorphosed mid-crustal blocks, particularly in spatial association with major crustal structures.

Orogenic gold deposit formation is inherent to juvenile crust formation throughout the Earth’s history. It is reflecting an evolution from dominantly greenstone-hosted deposits in the Precambrian to essentially identical deposits in metasedimentary host rocks in the Phanerozoic. Processes that controlled the generation of mineralizing fluids varied little over geological time and basically reflect synorogenic thermal events that mobilized fluids and metals during pro-grade metamorphism along active continental margins. However, evolving styles of plate tectonics on a cooling Earth, affected the overall preservation potential of orogenic gold deposits and controlled the temporal distribution of the ores (Golffarb and Groves, 2005).

Orogenic-type deposits are locally high grade and may yield large quantities of gold. While the vein systems or structures can be relatively easy to trace, ore shoots are less predictable.

9.0 EXPLORATION

The following is a summary extract from Leader et al., 2007 and Laudrum et al., 2008, to which the reader is referred for more detail.

9.1 CHOCO 10

Prior to Gold Fields acquiring control of the Choco 10 mine and property as a whole, exploration on the Choco concessions had included:

- Drilling.
- Line clearing and gridding.
- Construction of access trails.
- Excavation of exploration trenches.
- Surface geochemical surveys including stream sediments, soils and rocks at several levels of detail.
- Topographic and concession boundary surveying.
- Geological mapping at several scales and levels of detail.
- Airborne and ground geophysical surveys including frequency domain electromagnetics, magnetics, radiometrics and induced polarization.
- Structural interpretation using radar imagery.
- Metallurgical testing.
- Petrographic studies.
- Structural studies.

For the Choco 10 deposit, the outcome of all the above work was the delineation of the present deposits, Rosika, Coacia, Pisolita (RCP) and Villa Balazo-Karolina (VBK). The RCP deposits are currently being mined and the VBK deposit is scheduled to be mined in the near future. The exploration work carried out at the mine site by Gold Fields from March, 2006 included, primarily, infill drilling designed to advance resource classifications plus in-pit (to depth) and step-out drilling to expand resources.

From March, 2006 until the acquisition by Rusoro, Gold Fields, operating through El Callao Holdings, had completed 7,612 m of diamond drilling (39 holes) and 5,681 m of reverse circulation (RC) drilling (74 holes) within eight target areas. Other exploration activities include regional and local mapping aimed at resolving the regional stratigraphy and structural

setting, a detailed helicopter and fixed-wing airborne magnetic survey of the district, and development of technical strategic alliances with other companies including government agencies to enable sharing of data and knowledge.

Since Rusoro acquired the Choco 10 operations, they have continued to undertake a series of drilling programs to delineate potential ore extensions both horizontally and down the dip of each of the deposits as well as continue drill definition of the known mineralized zones to raise each of the deposit's Mineral Resource categories. A total of 406 diamond drill holes have been completed on the Choco 10 deposits since 2006.

Rusoro has also completed several geotechnical drilling programs within the deposits as well as other infrastructure and facilities. These programs were designed for open pit rock mechanics studies, condemnation drilling around the mill site, proposed waste dumps and tailings dam placement. Drilling and pit development in the mill site area was completed for infrastructure foundation studies.

9.2 INCREIBLE 6

Between 1991 and 1994, General Mining carried out a work program on Increible 6 that consisted of soil sampling on a 500-m spaced grid (600 km total), 105 trenches (2.5 km total) and 1,639 m of diamond drilling in 12 holes. This exploration was concentrated in three areas: Payara, Nuevo Rosario (which includes Olga-Enoc, Ingrid-Elisa and Cristina and Culebra deposits), and La Ramona. At Payara and La Ramona the work identified a series of narrow quartz veins and small miners' workings.

In 1994, Golden Star began its exploration program on Increible 6. The company re-examined the existing data, and then completed a soil and auger program (658 samples), trenching, geophysics, geological mapping, and 2,479 m of drilling in 19 holes. Almost all of the advanced work was concentrated on the east-west trending quartz veins and shears in the Nuevo Rosario area. Golden Star left in 1995 without earning any interest in the project.

There was no exploration between 1995 and 2004 on the concession.

In 2004, Rusoro began exploration and drilling on the concession. The first drilling was completed in September, 2004 utilizing one diamond drill. At the time of the 2006 Technical Report on Increible 6 by Scott Wilson RPA (Scott Wilson RPA, 2006), approximately 43,000 m of drilling had been completed.

Since 2004, Rusoro has concentrated most of its efforts on a series of extensive drilling programs. In addition to the RC and diamond drilling, other work has included small programs of line cutting, soil sampling, and petrographic work. Surface surveying included a detailed topographic survey over the main Ingrid-Elisa area allowing for the preparation of an updated 2-m contour topographic map.

10.0 DRILLING

10.1 CHOCO 10

The following has been reproduced from Leader et al., 2007.

“Delineation of mineralization at Choco 10 has been accomplished by means of drill holes which have traced the gold mineralization from surface to a vertical depth of between approximately 200 m (Pisolita) and 450 m (Rosika/Coacia). The rock package has been subjected to intense tropical weathering and has been weathered to saprolite near surface.

“Most of the drilling to date has focused upon the definition of surface mineable resources but deeper drilling has indicated that potential exists for underground development on some of the mineralization. The mineralization is open to depth at Rosika/Coacia and at VBK. At Rosika/Coacia, the potential for ore development within hanging-wall gabbro is also being evaluated. Other targets have been identified proximal to the current reserves which may provide additional resources once drilled.

“In most instances, the drilling penetrates the mineralization at an angle other than 90° requiring correction factors to be applied for true thickness determinations which are enabled by core descriptions, high core recoveries and down-hole survey data.

“The drill hole database utilized to generate the current resource is the result of a number of drill campaigns conducted over the history of the Choco 10 deposit.

The present drill hole database covers all four known deposit (Rosika, Coacia, Pisolita and VBK). There have been three drill methods used through the exploration of these deposits (diamond, reverse circulation and aircore). Diamond drilling has been the only method used since 2007. These methods have been described in previous NI 43-101 compliant reports submitted by Micon (2003) and Mine Development Associates (MDA, 2005) and are not repeated below.

The up-to-date drill statistics are as listed in Table 10.1.

Table 10.1
Choco 10 Deposits – Drill Summary

Drill Type	No. of Holes	Length (m)	Percentage
Aircore	1,198	43,368.09	16.06
Diamond Drilling	650	167,460.23	62.00
Track Drill	3	1,335.40	0.49
Reverse Circulation	611	57,902.60	21.44
Track Reverse Circulation	2	30.00	0.01
TOTAL	4,464	270,096.32	100.00

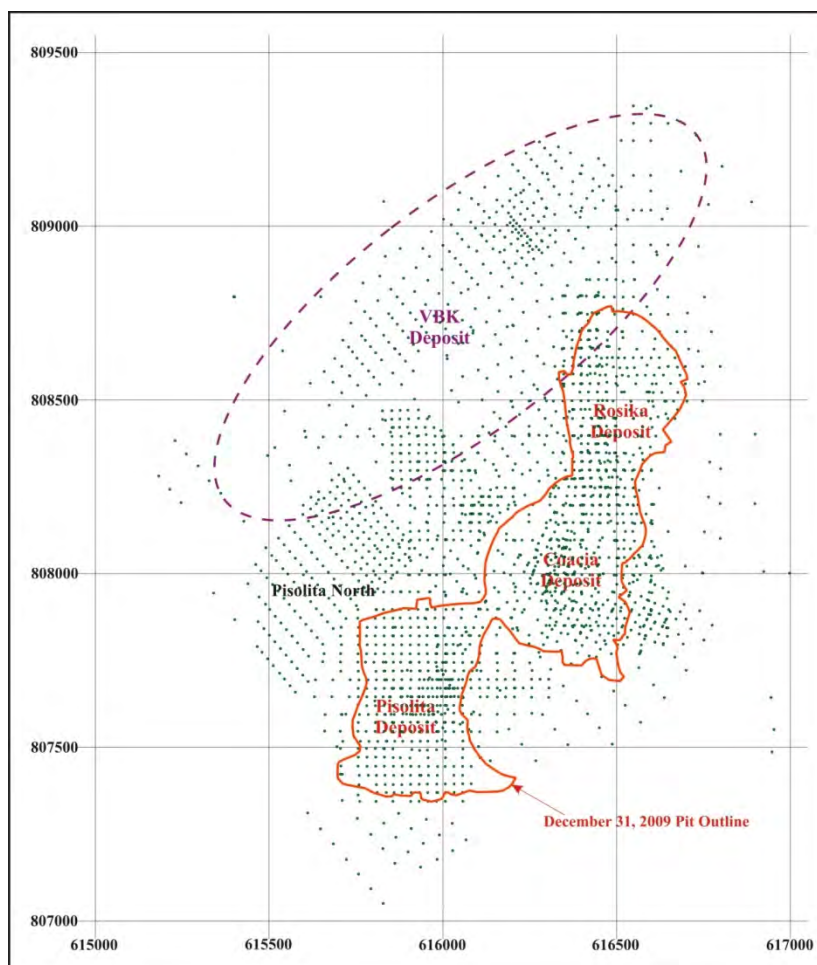
Additional drilling since the last mineral resource estimate totals 406 diamond drill holes which equates to 45,453 m. The focus of the drilling is illustrated in Table 10.2.

Table 10.2
2007 to 2009 – Drill Summary

Drill Type	No. of Holes	Length (m)	Percentage
Pisolita	227	17,720	39
Coacia	76	7,442	16
Rosika	68	7,046	16
VBK	35	13,245	29
TOTAL	406	45,453	100

This additional drilling focused primarily on the VBK and Pisolita North area. Figure 10.1 illustrates the complete drill hole database collar locations with respect to the deposits and the current open pit.

Figure 10.1
Choco 10 Drill Hole Collar Locations



10.1.1 Core Statistics

Drill hole spacing varies within each deposit as well as between deposits. The spacing ranges from 12.5 to 12.5 m to 50 by 100 m. The majority of the holes cut perpendicular to the strike of the target vein systems however they rarely intersect the vein systems at a true width. There are a minor number of holes that are vertically drilled.

The drill programs have explored the deposits to a maximum vertical depth of approximately 625 m below the topography.

The preferred drill type used on the Choco 10 deposits is diamond drilling which accounts for 62% of the total metres drilled to date. Aircore and reverse circulation holes represent 16% and 21% respectively the other major types used on the concession.

The majority of the diamond drill holes were drilled with HQ-size (63.5 mm) equipment. The holes were reduced to NQ-size (47.6 mm), if conditions deteriorated with depth or if wedging was required from the parent hole. PQ-size (85.0 mm) equipment was used for drilling in softer saprolitic/saprock material which was then cased when fresh hard rock was encountered. The drill stem was then reduced to HQ.

Core recovery at Choco 10 has historically been very good. Recovery data for diamond drilling have been systematically collected throughout all recent drill campaigns since 2003. The core recovery and associated RQD data are listed in Table 10.3.

Table 10.3
Drill Statistics

Years	Measurements	Core Recovery	RQD
2003 - 2006	41,767	94.54	89.45
2006 - 2009	13,048	97.18	94.91

10.2 INCREIBLE 6

The following has been reproduced from Laudrum et al., 2008.

“All drilling on the Incredible 6 gold project is completed by independent contractors. A number of commercial drilling companies operate in the El Callao district including;

- 1) “Perforaciones MayorTec (subsidiary of Major Drilling – New Brunswick, Canada): approximately 20 diamond drills operating in country since 1991, completed drilling at the Incredible 6 project between 2004 and 2006.
- 2) “Core Biel Drilling (Puerto Ordaz, Venezuela): seven diamond drills operating in country since 2002 (previously St. Lambert Drilling operating since 1992), completed drilling on the project between 2006 to the present.
- 3) “Perforaciones Caroni (Puerto Ordaz, Venezuela): four diamond drills, operating in country since 1996, completed limited drilling on the project in 2005.

- 4) “AK Drilling: four RC drills operating in country since 2004, completed drilling on the project between 2006 to the present.

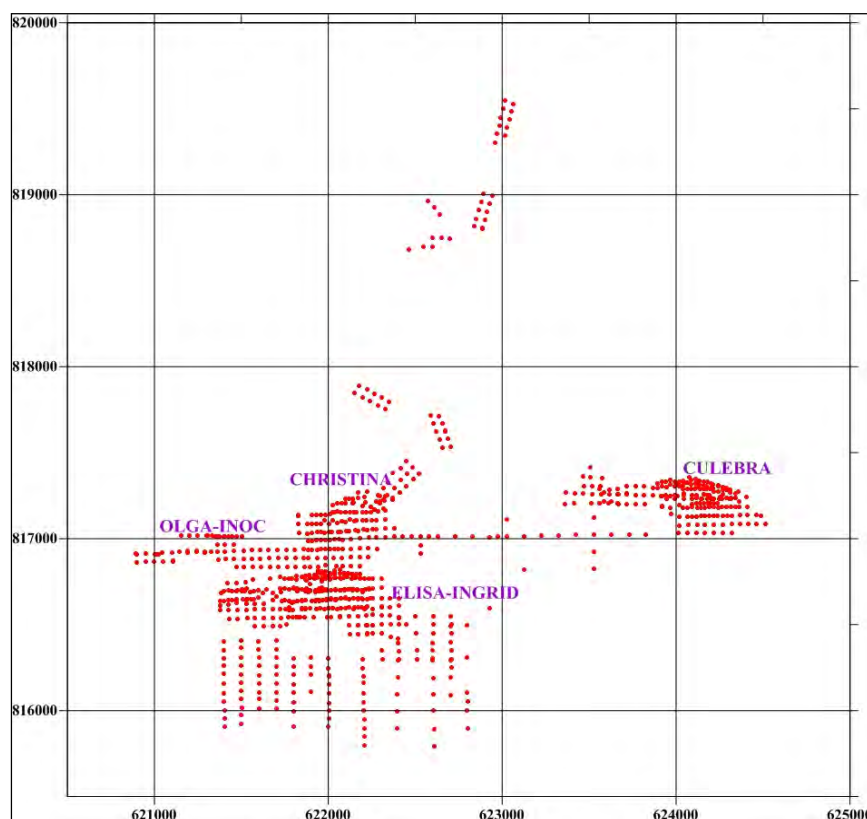
Along the main East-West mineralized trend (Nuevo Rosario) the diamond drilling has been done on approximately 50 m by 50 m pattern. The RC drilling has been used as in-fill drilling with respect to the diamond drilling. In places the drill pattern has been reduced to 25 m by 25 m.

Diamond drill core sizes were typically HQ which would be reduced to NQ if conditions warranted it.

Continuous drilling of the Incredible 6 deposits has been completed, until 2008-9, in order to expand and in-fill Nuevo Rosario. Drill programs also tested, anomalies away from this main East-West mineralized trend.

Within the Incredible 6 drill database there are 519 diamond drill holes with a combined length of 110,064 m and 472 reverse circulation holes with a combined length of 76,957 m. Therefore the total number of holes completed on the concession is 991 holes totalling 187,021 m in length and is illustrated in Figure 10.2.

Figure 10.2
Incredible 6 Drill Hole Collar Locations



10.3 CORE LOGGING AND SAMPLE PROCEDURES

10.3.1 Choco 10

A detailed description of the sampling programs at the Choco 10 deposits up to 2006 can be obtained from Leader et al., 2007.

Since the acquisition of the Choco 10 property by Gold Fields and more recently by Rusoro, an aggressive and systematic exploration program has been implemented. The exploration programs have been completed to industry best practice.

A series of 25 procedures and associated forms were originally developed by Gold Fields to cover all aspects of exploration, drilling, sampling, assaying and resource modelling at the Choco 10 concession. The procedures cover both reverse circulation and diamond drilling. Many of the procedures include diagrams and required forms to be filled out at various stages of the process. They cover all aspects of drilling from a new request for drill holes, sampling, core density, mapping, logging, database entry, QA/QC, resource modelling and reconciliation. Rusoro has adopted these procedures in their operation throughout Venezuela. Details regarding these procedures can be obtained by examining the individual procedures listed in Table 10.4.

Table 10.4
Rusoro Geological Procedures

Number	Description
MRM_M001	Geological Legend Manual
MRM_P002	Drill Request
MRM_P004	Reverse Circulation – Drilling, Sampling, QC and Logging
MRM_P006	Diamond Drilling
MRM_P007	Diamond Core Management – Core handling, geotechnical logging, photos, sampling
MRM_P008	Diamond Geological and Structural Logging
MRM_P010	Core Density Data Collection
MRM_P012	Reverse Circulation Grade Control – Drilling, sampling, QC and logging
MRM_P013	Open Pit Mapping
MRM_P014	Sample Consignment
MRM_A007	Laboratory Audit Checklist
MRM_P016	Rock Sample Preparation & Analysis - Exploration
MRM_P017	Rock Sample Preparation & Analysis – Mine Grade Control
MRM_P018	Database – Data Entry
MRM_P019	Database – Data Validation
MRM_P020	QC Sample & Data Management
MRM_P021	Resource Modeling – Geological Interpretation
MRM_P022	Resource Modeling – Geostatistical Analysis
MRM_P023	Resource Modeling – Estimation & Classification
MRM_A001	Confidence Limits for Resource Classification
MRM_P032	Reconciliation
MRM_P034	Rock Sample Check Analysis - Exploration
MRM_P035	Mine Planning
MRM_P037	Rock Sample – Cyanide Leach Testwork
MRM_P039	Blasthole Sampling

The protocols listed above have assisted the Geology and Exploration departments to obtain reliable data and to assist resource modeling and mine design. An example of the detail within each of these procedures is provided below.

Procedure MRM_P006 is the diamond core management protocol which provides details on each step involved in the handling of drill core upon arrival at the core yard. It is summarized below:

- Arrival of core boxes at core yard.
- Re-align core, metre marks and measurement of recovery and RQD.
- Photograph core.
- “Quick” geological log of core.
- Core cutting lines marked to bisect the dominant fabric.
- Selection and marking of sample intervals. These are constrained to geological units. The minimum length sample for HQ core is 0.25 m and maximum sample length is 1.5 m. Internal to geological units the preferable sample length is one metre.
- Flag and tag sample sites with cognizance of QC sample sites.
- Move core to cutting area.
- Prepare sample bags based on sample tags observed in boxes.
- Saw core, ensuring consistent half goes in bag, other half back in box.
- Samples are moved to sample preparation area and organized for shipping.
- QC samples are inserted into the sample stream.
- Samples are dropped off at the laboratory with requisition prepared by database administrator.
- Core boxes are moved back to core shack for detailed geological logging.

All diamond drill core, RC cuttings, pulps and rejects are stored in secure compounds either in La Ramona (El Callao – Rusoro’s Exploration Mine Office) or Tumeremo (Rusoro’s General Exploration office). Both compounds have perimeter wire fences, locks and have a 24 h guard as part of the security protocols.

Before 2007, there were multiple data validation reviews undertaken on the Choco 10 deposit to ensure high standards of quality. Independent reviews were undertaken by BHP

Engineering, Micon, Analytical Solutions Ltd., Smee and Associates, MDA and Maxwell. All review recommendations have been followed by Gold Fields and Rusoro.

It is Micon's opinion that the sample method and approach carried out on Choco 10 core and RC samples since April, 2006 have been conducted well within acceptable standards.

10.3.2 Incredible 6

For a complete account of the early sampling method and approach used by Rusoro can be found in previous Micon reports (Laudrum, 2008). The following has been reproduced from the Laudrum et al., 2008 report.

“Exploration on the Incredible 6 gold project has followed industry standard methods including the collection of stream sediments, soils, rocks (both float and outcrop), chip samples from trenches and roadcuts, and drill sampling.

“The level of sampling ranges widely within the project area. Several areas have detailed sampling, on surface and to depth through drilling, while other parts of the concession have been covered with only broad first pass sampling. A detailed review and analysis of all available information did not identify any factors that impact the accuracy and reliability of the data. All sampling is of good to excellent quality and the samples are considered to be representative and without sample biases.

Briefly, when drilling programs are being run, the geological staff is on site 24 h/d to supervise the logging and sampling procedures. When the core is delivered to the coreshack, specially trained core handling crews that:

- Open boxes.
- Lay out the core in proper order.
- Clean core, if necessary.
- Label metreage on core.
- Label boxes with metreages contained.
- Measure and record core recovery and rock quality determination (RQD).

The core is logged by the geologist as per the documented procedures on hardcopy which is transferred to computer at a later stage. The geologist identifies and marks the sample intervals on the core and in the hardcopy records. The respective samples are then marked on the core and in the boxes. A running sequence of consecutive numbers is assigned to each drill project and hole. The core handling crew assigns those numbers to the core samples and the inserted QA/QC blank/standard/duplicates as part of the sample stream. The diamond drill core is split in half by a diamond saw. One half is the assay sample and the second half is returned to the core box for permanent storage. Duplicate samples are made from the remaining half core. This is sawn into quarters with the remaining quarter being replaced in the core box. The drill samples and QA/QC samples are bagged with pre-numbered bags.

Micon on several site visits to the property have reviewed the sampling method and approach used by Rusoro over several years. The core has been examined during these site visits and interviews have been conducted with the geological team that was tasked to perform these duties. The list of procedure to follow through the entire program and using the Maxwell Database system makes operator error difficult. Micon has not identified any factors that would impact the accuracy and reliability of the data used in the mineral resource estimate.

10.4 SURVEYING OF DRILL HOLES

Surface drill hole collars are surveyed with Rusoro's survey department using a chain and theodolite. Downhole surveys are done with a Reflex Instruments downhole survey tool. A survey is taken nominally at every 50 m down the hole.

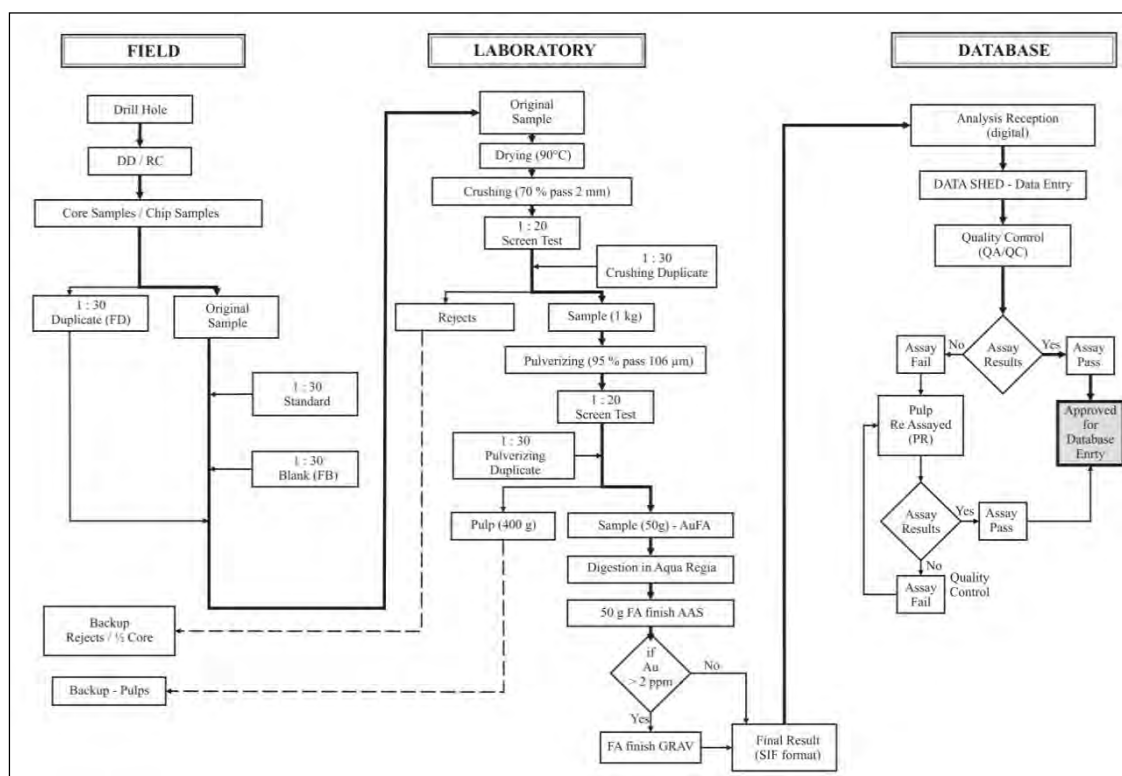
11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

A detailed description of the sampling preparation, analysis and security up to 2006 can be obtained from Leader et al., 2007, Laudrum et al., 2008.

11.1 SAMPLE PREPARATION AND QA/QC BEFORE DISPATCH OF SAMPLES

Sample preparation and analysis from 2007 to the present have strictly followed the procedures originally implemented by Gold Fields for all exploration drill samples (MRM_P016 Rock Sample Preparation and Analysis – Exploration) and for mine drill samples (MRM_P017 Rock Sample Preparation and Analysis – Mine Grade Control). Figure 11.1 illustrates the entire sampling, assaying and reporting process.

Figure 11.1
Sample Flowsheet for Choco 10 Drill Samples



All samples are categorized as regional exploration, resource definition or grade control samples. There are slight modifications to the sample preparation and assay process for the different categories and is documented in Table 11.1.

Generally, all samples are dried at a temperature not exceeding 110 °C. The total sample is crushed to 90% passing -2 mm (10 mesh) and then riffle split to produce a 750 g to 1,000 g sub-sample. Pulverization of the sub-sample is to at least 95% passing -106 µm (150 mesh). A 400 g pulp sample is retained and returned to Rusoro for permanent storage. Analysis is

completed by fire assay using an Atomic Absorption Spectroscopy (AAS) finish, with an additional gravimetric finish for all results exceeding 2 g/t gold. This process is illustrated in Figure 11.1 which also shows the frequency of the submission of all quality control samples, including field duplicates, standards, blanks, sample preparation duplicates and analytical duplicates.

Table 11.1
Rusoro Sample Preparation and Assay Process

Lab Procedures	Regional Exploration	Resource Definition	Grade control
Max Oven Temp	70°	<110°	110°
Pulverization	750 - 1,000 g		500 g
Fire Assay	50 g		30 g
Gravimetric	>2 g/t		1 in 20 of >2 g/t

11.1.1 Sample Preparation, Logistics and QA/QC

The quality control (QC) sample insertion protocols employed by Rusoro are summarized in Table 11.2 and illustrated in Figure 13.1.

Table 11.2
Rusoro Quality Control Protocols

QC Sample	Exploration		Mining
	Regional	Resource Definition	Grade Control
Blanks	1 in 20	1 in 30	1 in 40
Standards	1 in 20	1 in 30	1 in 40
Field Duplicates*	1 in 30	1 in 40 DD, 1 in 30 RC	1 in 40
Lab QC	Exploration and Mining		
Preparation Duplicates	1 in 30		
Analysis Duplicates	1 in 30		
Preparation Passing	Minimum 70% passing 10 mesh (-2 mm)		
Analysis Passing	95% passing 150 mesh (-106 um)		
Screen Tests	1 in 20		

* 'Geologically random'. Diamond core prep duplicates - ½ core for Resource Definition. ¼ core for Regional Exploration.

Occasionally quality control samples become contaminated or degrade over time and hence fail to represent a control sample. Rusoro has a system in place to identify quality control samples that fail. Table 11.3 lists the control samples that are used in the system and defines the individual failure criteria.

Once a control sample has been identified as being a failure, the appropriate action is documented in Table 11.4.

Table 11.3
Rusoro QC Sample Failure Criteria

Gold Fields Venezuela - Quality Control Table of Logic			
Rule	QC Type	QC Measure	Description
1	Blanks (Standards)	Contamination	A blank greater than 5 times the mean (baseline) is a failure (25 ppb SGS, 50 ppb Triad)
2a	Standards	Accuracy	A standard greater than 2 standard deviations from the mean is a failure
2b	Standards	Accuracy	Grade Control - A standard greater than 3 standard deviations from the mean is a failure
3	Standards	Accuracy	2 adjacent standards greater than 2 standard deviations from mean on same side of mean is a failure (indicates bias)
4	Field Duplicates*	Precision (field + lab)	% Difference >20% to be investigated, HARD of >10 is failure
5	Laboratory Duplicates*	Precision (prep + analysis)	% Difference >20% to be investigated, HARD of >10 is failure

* Duplicates <20 ppb Au (SGS) and <100 ppb Au (Triad) not included as failure as close to detection limit (1 ppb (20x) and 10 ppb (5x) respectively).

Table 11.4
Rusoro QC Failure Action Plan

Action No.	Description*
Action 1	If both rules 1 and 2a are broken for <u>all</u> QC samples in one batch, re-send entire pulps for re-assay
Action 2	If either rules 1 or 2a and rule 5 are broken for <u>all</u> QC samples in one batch, re-send entire pulps for re-assay
Action 3	For any 1 rule broken for individual GF QC samples only (except rule 4), if >100 ppb Au Triad/50 ppb Au SGS, send sample and adjacent samples until the next passing control sample is reached for re-assay of pulps
Action 4	Accept results if rule 4 broken (probable natural variation)

* No actions are taken for Mine Grade Control samples except under specific circumstances.

11.2 LABORATORY SAMPLE PREPARATION, AND ANALYSES

Although Rusoro has an assay laboratory next to the Choco Mill, no exploration samples are assayed at this facility. Some grade control samples from the mine were assayed at the Rusoro Mill laboratory in late 2008 to early 2009 but the assays are not part of the database used in the Mineral Resource estimate.

TRIAD laboratory in El Callao was initially used exclusively by the previous operators of the Choco 10 concession. Due to concerns centred around quality control and assay turn-around times, particularly with regard to grade control samples from the mine, Gold Fields in 2006 started contracting ACME Analytical Laboratories Ltd. located in Guasipati (20 km north of El Callao) to prepare and assay samples. Since that time Rusoro has used other accredited labs to handle the volume of samples in a timely manner. All laboratories are internationally accredited with ISO 9001.

Table 11.5 lists the laboratories used from 2007 to 2010.

Table 11.5
Rusoro Drill Hole Sample Laboratories

Sample Preparation			Sample Analysis		
Laboratory	Location	Number	Laboratory	Location	Number
TRIAD	El Callao, Ven.	79	TRIAD	El Callao, Ven.	79
Acme Analytical	Guasipati, Ven.	437	Acme Analytical	Guasipati, Ven.	354
			Acme Analytical	Santiago, Chile	37
			SGS Venezuela	Caracas, Ven.	46
SGS Venezuela	Caracas, Ven.	15	SGS Venezuela	Caracas, Ven.	15
Actlabs	Tumeremo, Ven.	29	Actlabs	Tumeremo, Ven.	29

11.2.1 Density Analysis

Since 2006, regular density measurements have been taken for 10-15 cm core lengths for every interval of a different weathering unit, rock or alteration type according to Rusoro's protocol MRM-P010:

- Water immersion is undertaken on all samples, avoiding direct measurement of the volume of water displaced by a sample. Samples are dried at 100 °C for 24 h then weighed to determine the dried mass. The sample is placed on a support of known weight in water and the sample suspended in water. The dry bulk density is determined as the mass of sample in air divided by the difference between the mass of the sample in air and the mass of the sample in water.
- Weathered samples are dried as above, and then the sample is weighed to determine the dry mass. The density of the wax is determined. The sample is then completely sealed by a coating of hot wax. After cooling the sample is weighed again in air. The sealed sample is then suspended in water and then weighed. The dry bulk density is determined as for rock samples, taking into account the volume of the wax.

The mine and exploration departments have completed extensive density testing on the various rock types and the weathered zone throughout the Choco 10 property. The density values are determined using raw data which has been interpreted and modified where necessary. The latest Competent Person's Report states "The main changes involved a decrease in saprolite density from 2.2 t/m³ in the 2006 resource model to between 1.60 and 1.90 t/m³ in the current resource; an increase in saprock density from 2.2 t/m³ in the 2006 resource to between 2.3 and 2.8 t/m³ in the current resource and an increase in fresh rock densities from 2.6 t/m³ for all rock types in the 2006 resource to between 2.7 and 2.9 t/m³ in the current resource" (Gold Fields, 2007). A detailed discussion on the density values can be found in the Competent Persons' Report, Appendix 3. Rusoro uses the following the conservative densities which have been extensively tested:

- Saprolite is 1.60 t/m³.
- Saprock is 2.35 t/m³.
- Fresh rock is 2.70 t/m³.

11.2.2 Security

The security of geological samples is paramount to have assured integrity of a geological database and subsequent mineral resource/reserve estimates. Rusoro has a strict chain of custody procedure for all aspects of sample handling and storage. The procedures for samples being shipped to assay labs is as follows:

- Normally, Rusoro's geological staff collects and transports the RC chip bags or drill core boxes from the field drill sites to Rusoro geological facilities in either La Ramona (El Callao) or Tumeremo. A chain of custody list is created in the field. When the samples arrive at the compound facility the samples are checked off from this list and put into temporary storage awaiting geological logging and sampling.
- Rusoro's geological staff creates the samples for assay at the geological facilities. Sample bags are tied at the top with a cloth tie. Then, for transport, a maximum of 5 samples are placed in a large plastic bag. One sample tag/ticket is placed in the sample bag (bag identified in permanent marker on the outside), and the second tag/ticket comes to the office for QC and sample mixing checking. These second tags are kept, and destroyed (burnt) only after 6 months, in the case they are needed as part of any quality control issues. Exploration tags are kept forever.
- MRM_P014 covers the laboratory sample consignment procedures. The main focus of this procedure is to develop the Sample Consignment Form. The form is part of the Maxwell DataShed software program which requires all compulsory information with respect to the sample(s) being assayed, to be logged into the system correctly and completely. The sample consignment form is signed at the Rusoro geological compound prior to shipping to the laboratory(s). The samples are transported to the laboratory(s) by Rusoro staff or laboratory staff. At the laboratory the samples are checked and signed for by the laboratory staff. The sample forms are in duplicate; Rusoro keeps a copy and the laboratory keeps the other. The form is illustrated in Figure 11.2.

Figure 11.2
Sample Consignment Form

SAMPLE CONSIGNMENT FORM

Gold Fields Venezuela
Mina de Oro Choco 10

Create New Submission

Total_NoSamples: 195

Despatch_no: ECH000001 LabCode: TRIAD_ElCallao Despatch_by: Nancy_Aguilera Location: L01/L04

Prep_LabCode: SGS_Tumeremo Despatch_date: 19-May-04 Tomorrow Project: Choco 09 Purpose: Regional Geochemistry

Special_Instructions:
Oven dry ALL samples to max temp of 70°C. Entire sample is to be crushed to minimum of 70% passing -2mm. 1kg split from the crushed material is to be pulverised to 95% passing -106µm and a 400g sample collected. An additional split sample is to be taken 1 in 30 as prep duplicate, and an additional from the pulverised material 1 in 30 as an analytical duplicate.

SampleID From	SampleID to	Sample Count	Sample Type	Prep Code	Au Analysis Code	Me Analysis Code
00951	00971	21	Soil	DRY10 + SCRE	FAA 505	ICM12B
00972	00990	19	Soil	DRY10 + SCRE	FAA 505	ICM12B
00991	01000	10	Soil	DRY10 + SCRE	FAA 505	ICM12B
19001	19014	14	Soil	DRY10 + SCRE	FAA 505	ICM12B
19015	19035	21	Soil	DRY10 + SCRE	FAA 505	ICM12B
19036	19050	14	Soil	DRY10 + SCRE	FAA 505	ICM12B

Record: 1 of 10

Preview Report Save and Send CLOSE

The samples follow a chain of custody procedure at the laboratory as part of their ISO 9001 accreditation requirements. If samples are only prepared at the local laboratory and shipped to another laboratory for assay, a chain of custody procedure is followed throughout the process, again as part of their ISO 9001 accreditation requirements.

Rusoro's geological compounds (La Ramona (El Callao) and Tumeremo) have perimeter wire mesh fences that are locked after hours. A security guard logs people and samples in and out of both compounds. The La Ramona compound is utilized by several non-geological staff. The core facility within the compound has no perimeter wire mesh fence and can't be considered totally secure. The core facility at the Tumeremo compound has a high wire fence inside the main warehouse building that is locked to all but required geological staff.

It is Micon's opinion that the sample preparation and analytical procedures carried out on Choco 10 and Incredible 6 core and RC samples since April, 2006 have been conducted well within acceptable standards, and that sample security is appropriate.

12.0 DATA VERIFICATION

12.1 SUMMARY

In addition to a comprehensive and very well documented set of procedures and internal QA/QC checks, Gold Fields and Rusoro has had external audits carried out on most aspects of the QA/QC and resource estimation process. Recent external audits include the following:

- Snowden, June, 2007, “Assessment of effectiveness of MRM Internal controls within Goldfields operations” (this report was commissioned by Price Waterhouse Coopers regarding compliance with the Sarbanes-Oxley Act 2002).
- Snowden, March, 2007, “External Resource and Reserve Audit”.
- Maxwell Geoservices, July, 2006, “Data Audit and Validation”.
- Smee and Associates Consulting Ltd, July, 2006, “A review of Quality Control Procedures and Field Sampling Techniques”.

The overall tone of the reports by external auditors has been very favourable, and it appears that significant recommendations from those reports have been implemented by Gold Fields and carried on by Rusoro.

Due to the numerous and rigorous procedures initiated by Gold Fields and continued by Rusoro, it has ensured that all data entered into the drill database, whether the samples are from air core, reverse circulation or diamond drilling passes the stringent verification, validation and QA/QC requirements in place at the Choco 10 and Incredible 6 deposits. An evaluation of the reliability for all drill data is included in the Competent Person’s Report, (Exploration and Quality Control) (Gold Fields, 2007).

Apart from observations of field and sampling procedures made during the site visit, Micon has not completed additional data verification for this report. Industry best practices are being followed and there have been numerous recent audit programs as mentioned above. In addition, Rusoro has adopted Gold Fields’ Mineral Resource Quality Management System in its entirety. Micon accepts that the data are verified and can be relied upon.

A synopsis of the historic data validation and audit programs that have been undertaken on the Choco 10 and Incredible 6 concessions can be obtained from Leader et al., 2007 and Laudrum et al., 2008.

12.2 DATA VALIDATION AND DATABASE

An aggressive and systematic exploration program of the Choco 10 concession was implemented by Gold Fields and continued by Rusoro at Choco 10 and Incredible deposits. The program and its subsequent database are based on industry best practice guidelines. The Gold Fields’ Mineral Resource Management (MRM) department initially established a framework of procedures, audits and sign-off documents for all key elements that input into

the generation of the database and all future resource models. The key components of the framework include:

- Validity – Controls to assure the validity of key activities.
- Accuracy – Controls to establish the accuracy of data inputs and outputs.
- Completeness – Controls to ensure the completeness of the process followed.
- Timing – Preventative and detective controls to identify potential risk and deviation of quality.
- Segregation of Duties/Sign-off – key members of the senior team responsible for aspects of the MRM process.

The list of procedures is illustrated in Table 11.1 and is not repeated here.

A database scheme was developed to provide a procedure to uniformly capture and securely store geological data, enabling straightforward access to all users. In 2007, Maxwell Geoservices was engaged to integrate the Bolivar Gold exploration and grade control databases into the Gold Fields Venezuela DataShed-SQL database (Maxwell 2007a). Rusoro adopted the Gold Fields methodology and procedures, when Rusoro took over the Choco 10 concession. They also utilized the methodology procedures for all their operations including Incredible 6. Rusoro over time has done some minor modifications or streamlining of the process which has not affected the integrity of the data stream.

The integration of the drill information involved data integrity checks and migration of validated data into the final database. All data that failed the data integrity checks were quarantined and assigned to a database administrator/ responsible geologist. This geologist was responsible to re-assaying the sample(s) until the conditions in Table 13.3 were met. He re-entered the data into the database and signed off on the change.

All geological data, from mining, resource definition and regional exploration, is stored within the centralized SQL Server database. Customized data entry forms have been developed in MS Access to minimize or eliminate data entry errors. As the database is relational, no codes can be entered that do not exist in linked library tables. The DataShed interface optimizes speed and accuracy of data entry, easy user access, data validation and extraction of the data to other software.

A Data Entry Template (MRM_P018) has been developed and customized for the input of all geological and sampling data and is illustrated in Figure 12.1.

The Sample Consignment Template (MRM_P014) has been developed to handle the request for sample analysis and has been discussed and illustrated in Section 11.2.2.

Figure 12.1
Drill Hole Data Entry

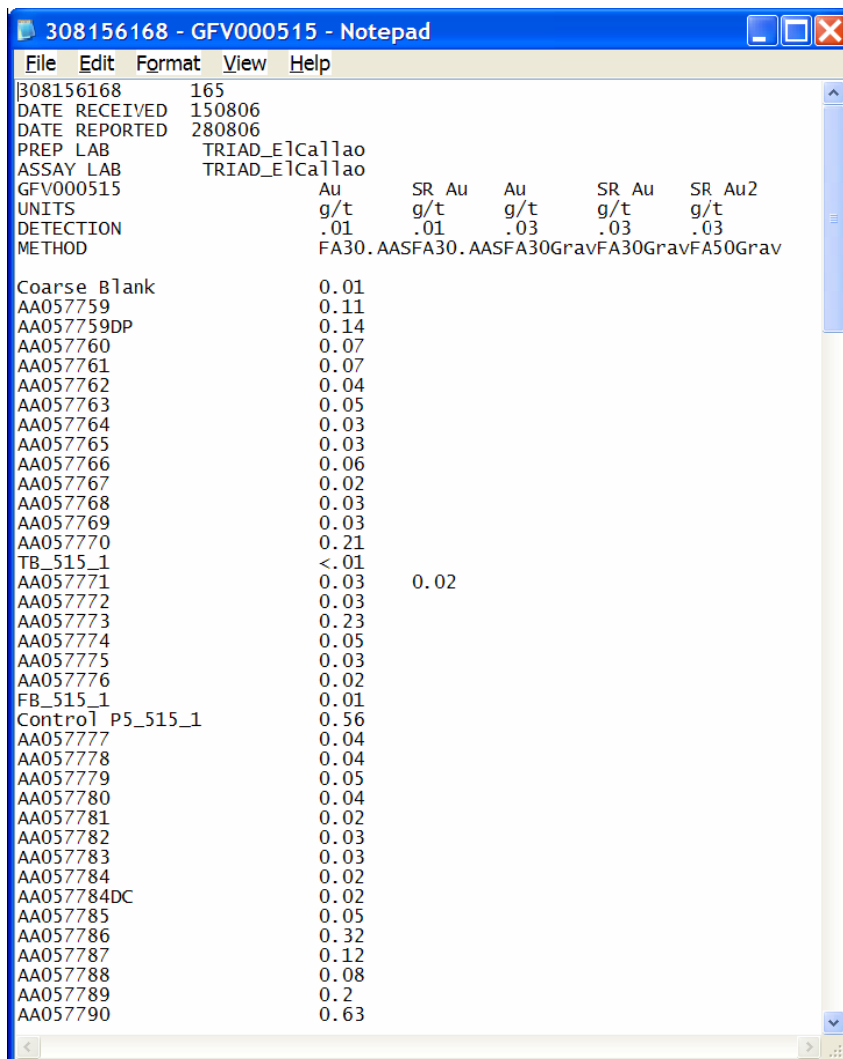
From	To	Mineral Code	Mineral (%)	Mineral Control	Mineral Style	Priority	Lith Grams	Mineral Timing
0	3.5 km		25 WT			1		
0	3.5 ka		0.5 WT			1		
0	3.5 ox		6 WT			2		
3.5	25.5 km		20 WT			1		

The corresponding assay results for the samples are automatically uploaded into the database in a text format known as a “sif” format file as illustrated in Figure 12.2. These files include detailed information about the batch, methods, units, detection limits and elements assayed. The file also includes all QC data in the sequence of analysis. The file cannot be tampered with, and must be generated in this consistent format by the laboratory. The assay data are stored in a normalized format to ensure all required information is stored for each sample, and that multiple assay results are stored for each sample. Import of assay data follows procedure MRM_P018, Database – Data Entry.

Data validation is controlled via rules, library tables, triggers and stored procedures. Data that do not meet defined rules on import are rejected and stored in buffer tables until corrected. Any data with a code not existing in a library table cannot be entered. Once all data for a drill hole have been entered into the database, the geologist responsible for the drill hole validates this against the original hard copy logs and signs off on it.

Data validation is incorporated in DataShed. The procedure runs queries against the database as illustrated in Figure 12.3. The validation includes checks for incorrect collar locations, testing for overlapping, missing or incorrect down-hole surveys, and incorrect collar location procedures and templates are available for all geological data. Data validation follows procedure MRM_P019, Database – Data Validation.

Figure 12.2
Assay Laboratory Sif File



Sample ID	Au	SR Au	Au	SR Au	SR Au2
UNITS	g/t	g/t	g/t	g/t	g/t
308156168	165				
DATE RECEIVED	150806				
DATE REPORTED	280806				
PREP LAB	TRIAD_ElCallao				
ASSAY LAB	TRIAD_ElCallao				
GFV000515					
DETECTION	.01	.01	.03	.03	.03
METHOD	FA30	AASFA30	AASFA30	GravFA30	GravFA50
Coarse Blank	0.01				
AA057759	0.11				
AA057759DP	0.14				
AA057760	0.07				
AA057761	0.07				
AA057762	0.04				
AA057763	0.05				
AA057764	0.03				
AA057765	0.03				
AA057766	0.06				
AA057767	0.02				
AA057768	0.03				
AA057769	0.03				
AA057770	0.21				
TB_515_1	<.01				
AA057771	0.03	0.02			
AA057772	0.03				
AA057773	0.23				
AA057774	0.05				
AA057775	0.03				
AA057776	0.02				
FB_515_1	0.01				
Control P5_515_1	0.56				
AA057777	0.04				
AA057778	0.04				
AA057779	0.05				
AA057780	0.04				
AA057781	0.02				
AA057782	0.03				
AA057783	0.03				
AA057784	0.02				
AA057784DC	0.02				
AA057785	0.05				
AA057786	0.32				
AA057787	0.12				
AA057788	0.08				
AA057789	0.2				
AA057790	0.63				

As part of the data validation, a procedure has been developed to enable original sample assay results which are determined to be unreliable to be replaced by validated re-analyses of those suspect samples (Quality Control). The “Assay Swapper” is illustrated in Figure 12.4. This procedure enables closing the loop on quality control within the confines of the database. A full audit trail with justification for the replacement is stored in the database.

Figure 12.3
Data Shed - Data Validation

Data Validation Tools

Queries in red typically relate to data required for statutory compliance when reporting resources and reserves.

Drillhole Collar and Survey | Drillholes Missing Downhole Data | **Validate Downhole Data** | Surface Samples

Select Validation

- ☒ Holes with duplicate Local Coordinates
- ☐ Holes with duplicate National Coordinates
- ☐ Holes with no or incomplete Local Coordinates
- ☐ Holes with no or incomplete National Coordinates
- ☐ Holes with no collar survey type
- ☐ Holes with null date drilled in Collar record
- ☐ Holes with no coordinates at all
- ☐ Holes with differing Original and National Coordinates
- ☐ Spare 9 - see your DBA to add a new report
- ☐ Spare 10 - see your DBA to add a new report
- ☐ Holes with no downhole Dip or Azimuth
- ☐ Holes with no surface Dip or Azimuth
- ☐ Holes with surveys below end of hole depth
- ☐ Survey records with Dip >90 or <-90
- ☐ Survey records with Azimuth <0 or >360
- ☐ Survey records with positive dips
- ☐ Spare 17 - see your DBA to add a new report
- ☐ Spare 18 - see your DBA to add a new report
- ☐ Spare 19 - see your DBA to add a new report
- ☐ Spare 20 - see your DBA to add a new report

VIEW DATA **PREVIEW REPORT** **CLOSE**

Figure 12.4
Assay Swapper

Assay Swapper

Enter Details for the Batch / Samples you wish to swap and choose a new category

Date From: 1/01/2007 Cal New Category for Swaps: SWAP

Date To: 13/02/2007 Cal

Lab: Triad Laboratory El Callao, Vene

Tables: tblDMSamp

DataSet: RCP_VBK

QC Category: PR

Swap By: ☐ Batch ☒ Samples

Next > Cancel

12.3 MICON DATA VERIFICATION

The QA/QC procedures, DataShed procedures and validation were reviewed by Micon's geological staff in 2007 and again on the 2009 site visit. Micon was able to observe RC chip sample logging and sampling, insertions into the sample streams of field duplicates, standards and blanks, and demonstrations of data entry and data control (such as follow-up procedures regarding control sample failures, database protocols, etc.).

The overall impression of the system is that Rusoro is expending tremendous effort (manpower and overall cost) to comply with all aspects of the Gold Fields system. The methodology is considered a very high standard for the industry. It adopts a rigid series of standards and procedures that attempts to eliminate all human errors throughout the collection and retrieval of geological information. Unfortunately, a large company like Gold Fields has the flexibility and manpower to incorporate a system like this, while a smaller company may find it difficult to comply with all aspects of the system.

It is Micon's opinion that Rusoro's system is working well. It is being operated by very competent professionals and technical staff.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTWORK

A Preliminary Assessment Study (PAS) for the Choco 10 expansion was issued in June 2009. The results indicated that an expansion to 20,000 t/d, and using the same leach/CIP flowsheet as currently employed, would produce the most favourable economics. At the time of the PAS, the plant was treating surficial deposits of predominantly highly weathered oxidised Saprolitic ore which is friable and can be processed at up to 7,500 t/d with recoveries of 90% plus. However, it was recognised that the plant throughput will decrease to about 5,000 t/d as deeper, more competent material is mined. The PAS considered an expanded plant treating 20,000 t/d of hard rock material to be optimal. This rate of throughput was taken as the basis for the Feasibility Study.

For the Feasibility Study, metallurgical testing of core was supplemented with fresh open pit material. Samples were sent to PRA in Vancouver and to Ammtec in Australia for comminution testing. The PRA tests concentrated on processing options and a limited amount of comminution work. The metallurgical processing routes considered during the PAS were investigated in depth, in addition to alternative process options. The main considerations examined were the incorporation of a gravity circuit for recovery of coarse and free gold and a sulphide flotation circuit with subsequent treatment of the concentrates. The flotation option indicated the possibility of slightly higher metallurgical recoveries. However, it was decided that the flowsheet currently in use by the 5,000 t/d operation should form the basis of the expanded plant, with some minor modifications. Inclusion of a gravity circuit was rejected on the basis that the disadvantages outweighed the possible advantage.

It is intended that the new 15,000 t/d facility will operate in parallel with the existing plant. To minimise duplication of facilities the plant layout was reviewed with particular regard to the crushing circuit. This “tie in” of the new and old plants was particularly important to ensure minimum disruption to production at the time. Both plants would use common facilities such as pre-leach thickening and tailings thickening and the gold recovery room. The elution circuits would be kept separate as gold on carbon loadings would likely be different and cycle times also different.

13.1 PREVIOUS METALLURGICAL TESTWORK AND FLOWSHEET DEVELOPMENT

Extensive testwork for the Choco deposit has been carried out over the last few years most of which, related to the oxidised saprolitic surficial deposits. However, mining has now progressed to the point where only small amounts of Saprolite remain and predominantly un-weathered ore will be treated in the future. Exhaustion of the Saprolite was anticipated and some testing was carried out on un-weathered material. As background information, previous testwork is described below.

A program of metallurgical testing was completed using samples characterizing each major mineralized zone identified at Choco. The purpose of the metallurgical testwork was to provide data to select metallurgical unit operations, develop an efficient process flowsheet and obtain design criteria for the process engineering and associated operating and capital cost estimating. The process design was based mainly on metallurgical tests conducted at

McClelland Laboratories, Inc (McClelland), Nevada in 1995. This testwork program used composite samples that represent the main zones of mineralization occurring at Choco. Additional metallurgical testwork programs include work conducted by Amdel Limited (Amdel), Australia in 1994 and at Lakefield Research Laboratories (Lakefield), Lakefield, Canada in 2003. The Amdel laboratory testwork was conducted on a range of samples representing near surface and weathered mineralization. The Lakefield program used individual mineralized samples of split drill core that represent a range of deposit mineralization. The objective of these tests was to confirm the results obtained from the McClelland work and to assess the potential variability of metallurgical results for different areas within the mining plan.

13.1.1 McClelland Testwork

Four composite samples representing the main mineralized zones were submitted to McClelland for testing in 1995. These samples were designated as shown in Table 13.1.

Table 13.1
Samples submitted to McClelland for testing in 1995

Sample	Description	Designation
CH01	Laterite	Surface
CH02	Clay/schist	Weathered
CH03	Silicified zone/quartz	Weathered
CH04	Volcanic bedrock	Fresh

The McClelland testing program comprised four stages. The initial stage included mineralogical examinations, Bond ball mill work index determinations and CIP cyanidation tests. The second stage comprised coarse bottle roll leach tests and the third stage consisted of heap leach column tests and agglomeration studies. The final stage of testing comprised detailed milling/cyanidation 93 studies and included abrasion index tests, CIP kinetic tests, slurry viscosity determinations and settling tests.

13.1.2 Amdel Testwork

The Amdel testwork was conducted on six samples and included coarse bottle roll cyanide leaching, gravity concentration, agitation cyanide leaching and Bond ball mill work index determinations. These tests provided an indication of the metallurgical performance, with regard to gold recovery, of the near surface mineralization. The six composite samples used in the Amdel test program were designated as shown in Table 13.2.

Table 13.2
Composite Samples used in the Amdel Test Program

Sample	Description	Designation
CL 1	Pisolitic laterite	Surface
PL1	Dicolite laterite	Surface
PV1	Clay/quartz vein	Weathered
CA1	Silicified/kaolinitic clay	Weathered
CSI1	Silicified alteration zone	Weathered
PA1	Geothitic clay, quartz veining	Weathered

Samples CA1, CSI1 and PA1 received by Amdel were too fine to undertake coarse ore cyanidation bottle roll tests, and CA1 and PA1 were too fine to determine Bond ball mill work indices.

13.1.3 Lakefield Testwork

The first phase of confirmatory tests conducted by Lakefield in May 2003 comprised Bond rod and ball mill work index determinations, and gravity and cyanidation gold recovery tests. The samples used for these tests consisted of composite samples CH05 and CH06, which represent fresh unoxidized bedrock from the southern and northern sectors of the Rosika deposit. A second phase of work at Lakefield included an additional 10 cyanidation bottle roll tests, undertaken on samples of fresh hard rock and soft, weathered mineralized material. A third series of tests conducted at Lakefield comprised 18 cyanidation bottle roll tests using standard conditions. These tests included 16 variability tests using individual samples of near surface material.

13.2 PREVIOUS TESTWORK RESULTS

13.2.1 Grinding

The McClelland, Amdel and Lakefield Bond ball and rod mill work index test results are presented in Table 13.3.

Table 13.3
Summary of Grinding Testwork Results

Laboratory	Sample	Work Index Type	Mineralization	Product Size P ₈₀ (µm)	Work Index (kWh/t)
McClelland	CH01	Ball	Surface	75	13.1
McClelland	CH02	Ball	Weathered	75	7.5
McClelland	CH03	Ball	Weathered	75	14.4
McClelland	CH04	Ball	Fresh	75	13.6
Amdel	CL1	Ball	Surface	72	5.4
Amdel	PL1	Ball	Surface	94	10.0
Amdel	PV1	Ball	Weathered	78	11.5
Amdel	CSI1	Ball	Weathered	70	9.7
Lakefield	CH05	Rod	Fresh	885	17.8
Lakefield	CH05	Ball	Fresh	78	14.1
Lakefield	CH06	Rod	Fresh	878	16.7
Lakefield	CH06	Ball	Fresh	80	13.7

13.2.2 Gravity Concentration

Gravity concentration tests were conducted at Amdel and Lakefield. These tests were undertaken to determine the potential recovery of liberated gold prior to cyanide leaching. The gravity concentration tests at Amdel were conducted using a Wilfley shaking table while the tests at Lakefield used a Knelson concentrator and Mozley mineral separator.

The gravity testwork results are summarized in Table 13.4.

Table 13.4
Summary of Gravity Testwork Results

Laboratory	Sample	Mineralization	Gold Recovery (%)
Amdel	CL1	Surface	30.0
Amdel	PL1	Surface	11.5
Amdel	PV1	Weathered	25.9
Amdel	CA1	Weathered	7.2
Amdel	CS11	Weathered	3.5
Amdel	PA1	Weathered	16.7
Lakefield	CH05/06 blend	Fresh	19.1

13.2.3 Cyanide Leaching

A summary of the McClelland testing results is presented in Table 13.5. This table includes results from 120 h coarse bottle roll cyanidation tests and 48 h conventional bottle roll tests conducted on all four samples. Results from 48 h agitated leach tests on CH01, CH02 and CH03 are also included.

Table 13.5
Summary of McClelland Cyanidation Testwork Results

Sample	Feed Size (80% passing)	Gold Head (g/t)	Gold Recovery(%)	Lime Added (kg/t)	CN Consumed (kg/t)
Coarse Bottle Roll Cyanidation Tests					
CH01	25 mm	1.58	86.9	14.6	0.41
CH02	25 mm	4.39	87.5	14.1	0.16
CH03	25 mm	2.81	65.8	4.1	0.15
CH04	25 mm	2.09	39.3	2.4	0.15
CH01	6.3 mm	1.65	91.7	14.8	0.46
CH02	6.3 mm	3.84	91.1	11.4	0.18
CH03	6.3 mm	2.23	80.0	3.8	0.14
CH04	6.3 mm	1.71	44.0	1.8	0.37
Conventional Bottle Roll Cyanidation Tests					
CH01	75 µm	1.75	94.1	16.0	1.00
CH02	75 µm	3.87	96.5	11.0	0.77
CH03	75 µm	2.69	91.9	4.2	1.09
CH04	75 µm	2.25	88.3	3.4	0.83
Agitation Cyanidation Tests					
CH01	75 µm	1.89	96.3	13.3	0.24
CH02	75 µm	3.39	96.0	4.9	0.30
CH03	75 µm	2.67	94.9	3.2	0.18

The results from the McClelland testwork program indicated that all of the different types of mineralization at Choco are amenable to conventional carbon-in-pulp (CIP) cyanidation at a

grind size of 80% passing 75 µm, although gold recoveries for sample CH04 (fresh ore) were typically lower than the other samples.

The McClelland coarse ore bottle roll tests indicated that the surface and weathered samples (CH01, CH02 and CH03) were also potentially amenable to heap leaching although CH03 was more sensitive to crush size with respect to gold recovery. Heap leach testing of the fresh ore composite sample (CH04) was not successful with regard to gold recovery, coarse bottle roll gold recoveries of less than 45% were reported with a fine crush size F₈₀ of 6.3 mm.

The gold kinetic extraction profile for the agitation leach tests performed on the surface and weathered ore samples, CH01, CH02 and CH03, suggested that the extraction was substantially complete in less than 24 h.

A summary of the Amdel cyanidation test results is presented in Table 13.6. This table includes results from 7-day coarse (-12.5 mm) bottle roll tests and 48 h agitated leach tests. The agitated leach tests were undertaken on ground samples of gravity test tailings. The comparative grind sizes used for the agitated leach tests were 80% passing 150, 106 and 75 µm.

Table 13.6
Summary of Amdel Cyanidation Testwork Results

Sample	Feed Size (80% passing)	Gold Head (g/t)	Gold Recovery (%)	Lime Added (kg/t)	CN Consumed(kg/t)
Coarse Bottle Roll Cyanidation Tests					
CL1	12.5 mm	2.47	86	6.2	0.8
PL1	12.5 mm	3.04	92	5.8	1.1
PV1	12.5 mm	8.30	66	5.2	0.9
Agitation Cyanidation Leach Tests (Gravity Tails)					
			Leach	G+L¹	
Surface – Ave ²	150 µm	2.02	95.50	97.00	1.9
Surface – Ave ²	106 µm	2.02	95.00	96.50	1.8
Surface – Ave ²	75 µm	1.94	97.50	98.50	1.9
Weathered – Ave ³	150 µm	6.85	87.50	89.50	2.0
Weathered – Ave ³	106 µm	6.52	94.00	95.00	1.9
Weathered – Ave ³	75 µm	6.54	92.50	93.50	1.9
Weathered – Ave ⁴	75 µm	5.02	93.00	93.75	1.6

1 Total gravity plus leach recovery

2 Average results from tests conducted on two surface samples, CL1 and PL1.

3 Average results from tests conducted on two weathered rock samples, PV1 and CSI1.

4 Average results from tests conducted on four weathered rock samples, PV1, CA1, CSI1 and PA1.

The relatively high gold recoveries achieved in the coarse bottle roll tests could partly be attributed to the considerable amount of comminution that occurred during the 7-day period. Nevertheless, these results indicate that the mineralization tested may be amenable to heap leaching.

The gold extractions achieved by the Amdel agitation leach tests were generally high. The average gold recoveries after 48 h leaching time, for material ground to 80% passing 75 µm, were 98% and 94% for surface and weathered material, respectively. The comparative leaching tests performed at different grind sizes indicated that gold recovery improved with a finer grind size for both surface and weathered material.

Generally the gold extraction during the agitation leach tests was complete within 24 h. The exceptions were the tests conducted on surface sample CL1 and the weathered sample CSI1, which both exhibited relatively slow leach kinetics.

The 48-h cyanide gold extraction was about 90% in all the Lakefield cyanidation bottle roll tests conducted on the fresh ore composite samples, CH05 and CH06. A gravity separation test using a bench scale Knelson separator realized 19% gold recovery, but this additional process step did not significantly impact the overall (gravity plus leach) gold recovery. The average results from these tests, together with the average of 10 bottle rolls conducted in the second phase of Lakefield testwork, and the average of 16 leaching tests conducted on samples of near surface material, are summarized in Table 13.7.

Table 13.7
Summary of Lakefield Cyanidation Bottle Roll Testwork Results

Sample	Feed Size (80% passing)	Gold Head (g/t)	Gold Recovery (% - 48 h)	Lime Consumed (kg/t)	CN Consumed (kg/t)
CH05/06 – Ave.	80 µm	2.97	90.0	0.54	0.35
Fresh – Ave ¹	70 µm	4.35	90.5	0.74	0.83
Weathered – Ave ²	66 µm	2.68	95.7	2.34	0.37
Surface – Ave ³	~35 µm	2.35	89.6	4.67	0.48

1 Average results from tests conducted on three fresh rock samples, CR03-12/13 and 17.

2 Average results from tests conducted on six weathered rock samples, CR03-19/20/24/24B/34 and 46.

3 Average results from tests conducted on sixteen surface samples.

13.2.4 Discussion of Results

The design metric Bond ball mill work indices selected for surface, transition and fresh ore, based on the testwork results, were 7.0, 12.0 and 14.0, respectively.

Gravity concentration tests suggested that a gravity circuit incorporated in the milling circuit might be able to recover up to 30% of the gold from certain feed types. However, there may be no net gain in overall (gravity plus leach) gold recovery by installing a gravity circuit prior to the cyanidation circuit.

The ground ore cyanide leaching testwork showed that all of the ore types tested were amenable to conventional cyanide leaching at a grind size of 80% passing 75 µm. For the majority of samples, the leaching kinetics was relatively good with the gold extraction generally complete within 24 h.

The gold extractions achieved during the ground ore cyanide leaching testwork were typically 90 to 98% for surface samples, 93 to 96% for weathered samples and about 90% for fresh samples. Lime consumption was generally high, moderate and low for surface, weathered and fresh mineralization, respectively. Cyanide consumption was relatively moderate for surface and weathered material but tended to be higher for fresh samples.

The results from the coarse ore bottle roll tests conducted by both McClland and Amdel indicated that the surface and weathered material could be amenable to heap leaching, although the fresh ore was not. However, considering the likely problems associated with the heap leach solution permeability of lateritic and clay type material, the process route selected for the recovery of gold from all types of material occurring within the Choco mineral resource was grinding followed by cyanide leaching and CIP.

13.3 TEST WORK BY INSPECTORATE AMERICA CORP. (PRA), 2008 - 2010

13.3.1 Sample Information & Test Procedures

Since the new plant would be treating predominantly harder competent rock as the overlying Saprolite would be mined out attention focused on those samples from the Preliminary Assessment work (reported June 3rd 2009) that met these criteria. The previous assessment work had included samples which were softer than future ore expectations and, where possible, these were disregarded for the Feasibility Study.

Samples used for the 2008 metallurgical testwork were identified by drill hole, sample label and type, as noted in Table 13.8. Samples 1, 2, 4, 6, 7 and 9 were characterised as “hard rock” and samples 3, 5 and 8 as saprolite.

Table 13.8
Summary of Samples Used in 2008-2010 Testwork

Incredible Comp. ID	Sample No.	Ore Type	Choco Comp. ID	Field Description	Ore Type
#1	152353	Hard rock	A1	VBK-01-D41HG	Ore shoots
#2	152354	Hard rock	B2	VBK-02-D41LG	Incl. halos
#3	152355	Saprolite	C3	VBK-03-FWMIN	FW surface
#4	152356	Hard rock	D4	ROS-01-MLODE	Alteration 2
#5	152357	Saprolite	E5	COA-01-MLODE	Alteration 2
#6	152358	Hard rock	F6	RCO-01-GABHW	Alteration 1
#7	152359	Hard rock	G7	COA-02-HWL0D	Alteration 1
#8	152360	Saprolite	H8	VBK-04-D410X	Oxide
#9	152361	Hard rock	J9	ROS-02-LODOX	Mixed oxide
			K10	COA-03-HWMIN	Oxide

To provide sufficient sample for detailed testing and optimise the number of tests, ores with the same characteristics were combined into a number of composites as noted in Table 13.9. Choco samples were described as underground, open pit (fresh rock) and oxide, referring to VBK (Villa Balazo-Karolina), ROS (Rosika), COA (Coacia), and RCO (Rosika/Coacia) lodes, foot- or hanging-wall origins. Each composite was homogenised and split into 1 kg test charges, an aliquot from each was submitted for gold fire assay, S- and C species and 30

element ICP-MS scans. In preparation for controlled flotation testing Bond ball mill work indices were determined and test grinds in a stainless steel rod mill were conducted to establish time vs. grind relationships.

Table 13.9
Composite Sample Identifiers

2008 Sample ID	New Equal Wt. Comp. Blend
VBK A1 + B2	MC-A
VBK C3 + H8	MC-B
RCO J9 + F6	MC-C
COA G7 + K10	MC-D
Hard rock #1 + #2	IC-1
Hard rock #6 + #7	IC-2
Hard rock #9	IC-3
Oxide	n/a

13.3.2 Cyanide Leaching

Bottle roll cyanide leaching tests using 1 kg samples were carried out at a pulp density of 40 wt% solids. Prior to adding sodium cyanide (NaCN), the alkalinity was adjusted with hydrated lime to pH 10.5 -11 and maintained at this level during the tests. NaCN levels were also maintained at specific targets and dissolved oxygen concentration monitored throughout the test. Solution samples were removed at 7, 24 and 48 h of retention time to determine silver dissolutions. Baseline tests were terminated after 72 h with filtration of pregnant leach solution (PLS). The solid residues were displacement-washed with hot cyanide solution, followed by two hot water rinses. The PLS and the final residue were analyzed for gold content by standard fire assay procedures.

The initial leach test conditions were, 1 g/L NaCN (Tests C1-C2) with target grinds of 74 µm (200-mesh). Test CIL-1 was similar but with addition of 20 g/L activated carbon, no kinetic sampling and 48-h duration. Grind times were doubled in Tests C3 and C4, both with overhead agitation, either with air or oxygen sparging for 48-h, but also maintained at 1 g/L NaCN. One leach test (GC1) was undertaken with a start NaCN concentration of 1 g/L NaCN but maintained at 0.5 g/L NaCN.

Reagent concentrations were determined using standard titration methods. CN-speciation, ICP and residue size assays were conducted.

13.3.3 Gravity Concentration

To determine if overall recoveries could be improved, single pass gravity separation tests were conducted in a laboratory centrifugal concentrator (Falcon) at grinds simulating likely cyclone underflow streams. Panning to produce smaller amounts of concentrate for assaying purposes and to also simulate further upgrading was employed.

Further tests on gravity concentration were also carried out but the method appears to offer no significant advantage over a whole-ore leach and so has not been used in the proposed flowsheet.

13.3.4 Flotation

Bench scale flotation testing was conducted in a Denver D12 laboratory flotation machine equipped with a metered forced-air supply. A 5 L cell size was used which enabled a 2 kg charge to be tested at 35% w/w solids, airflow was controlled manually to maintain the froth level. Collectors were added before aerating with timed conditioning periods. The primary flotation regimes were estimated based on existing industrial standards. The rougher product slurry of F1 only was filtered and weighed wet to permit a standard regrind to be carried out. Size assays were performed on the tailings from all tests.

Further tests on flotation concentration were also carried out but the method appears to offer no significant advantage over a whole-ore leach and so has not been used in the proposed flowsheet.

13.3.5 Bond Ball Mill Work & Abrasion Index Determinations

The Bond ball-mill work indices of a number of samples were determined by PRA using a Bico-Braun® laboratory mill and standard ball charge medium. Abrasion tests and SMC tests were undertaken at Hazen Research Inc. (USA) (Hazen).

13.3.6 Results and Discussion

The preliminary assessment work reported in the June 3, 2009 report provided useful background information but the primary focus of current discussions was centered on the results obtained at PRA for the Choco 10 project only. Preliminary diagnostic tests were conducted on few less extractable 2008 samples, to expedite the planned leach optimization on client-selected blends of the major material types

13.3.6.1 Head Analysis

Interval samples assayed and tested in 2008 had shown a persistent variation of gold grades, with a mean variance on the order of 20%. Fresh composites prepared in 2009 (Table 13.10) fell within the expected ranges, with a few outliers on the high-grade side when aggressive gravity scalping was pursued. Sulphate and silver levels were low (near detection limits), and traces of impurities of concern and few base metals were present. As implied by available project information, hydrothermal mineralization is mainly found within or at boundaries of more permeable structures, and cyanide leaching could be hampered by encapsulation, adsorption or “preg-robbing” via co-deposited carbon, clay, mica, spinels, sulphides or sulfo-salts, much more likely than in abundant quartz or non-porous carbonates, as indicated above.

Table 13.10
Head Assay Comparison of New Composites Tested

Comp. ID	Selected Head Assays, % or ppm								
	Au	As	Al	Ca	C(org)	S ⁼	Fe	Mg	Mn
MC-A	4.61	16	6.08	6.03	0.25	1.25	8.16	2.39	0.123
MC-B	1.94	8	5.88	4.31	0.23	0.91	8.58	2.33	0.149
MC-C	1.64	7	5.87	2.27	0.11	0.35	5.07	1.58	0.065
MC-D	1.19	<5	6.01	2.13	0.13	0.59	3.69	1.03	0.053
MC-Avg.	2.35	<9	5.96	3.69	0.18	0.78	6.38	1.83	0.098
IC-1	3.87	98	5.13	4.18	0.94	1.21	5.79	1.53	0.071
IC-2	2.76	985	5.92	2.06	0.15	0.43	1.47	0.45	0.020
IC-3	2.71	95	5.12	6.83	0.34	0.66	6.31	2.13	0.098

Hardness determinations indicated an average Bond ball mill work index (BWI) of 13.6 kWh/t at a 104 µm (150-mesh) closing screen, and a 3.35 mm (6-mesh) bulk S.G. of 1.994 (Table 13.11). Sample IC-3 was excluded as there was insufficient material. The data was consistent with those of previous suites of testing and confirmed a medium-hard character for most of the Choco-10 intervals submitted.

Table 13.11
Hardness and Bulk SG Data on New Composites

Parameters tested	MC-A	MC-B	MC-C	MC-D	IC-1	IC-2
BWI, kWh/t	12.5	11.0	12.9	14.4	14.2	16.6
Bulk SG	1.91	2.14	2.01	1.84	2.17	1.91

13.3.6.2 Diagnostic Testing

Better understanding of material characteristics in response to gravity scalping, and leaching with oxygen or various other amendments to baseline testing was considered prior to arrival of fresh materials for leach optimization studies. Head assays of the selected intervals are summarized in Table 13.12, for direct comparison with Table 13.10, above. Centrifugal concentrates were panned to differentiate various mineral classes, and ICP + cyanide-soluble gold assays were obtained (Table 13.13); the more refractory samples were then submitted for diagnostic leaching.

Table 13.12
Head Assay Comparison of Less Extractable Intervals

Interval ID	ppm			%					
	Au	As	Cu	Ca	Al	S ^T	Fe	Mg	Mn
MC-B2	7.27	<5	118	6.65	6.13	1.68	8.66	3.54	0.095
MC-D4	28.7	<5	57	3.61	7.49	1.92	4.10	1.47	0.052
MC-F6	3.03	<5	125	6.65	6.93	0.59	8.00	4.15	0.093
MC-K10	1.52	<5	40	0.07	9.80	0.01	3.31	0.65	0.066
IC-#1	4.35	70	105	4.61	5.31	1.85	7.78	1.75	0.078
IC-#6	2.92	1200	15	2.77	6.46	0.38	1.73	0.54	0.022
IC-#7	2.91	940	13	1.80	7.24	0.37	1.39	0.36	0.016
IC-#8	2.25	400	27	0.57	8.90	0.15	2.50	0.15	0.016

Table 13.13
Gravity Response of Less Extractable Intervals

Interval ID	Pan Assays				Pan Conc. Distributions, %				
	Au ppm	Mn/As %	Fe %	Ag ppm	Mass	Au	Mn/As	Fe	Ag
MC-B2	710	0.03	70.0	67.4	0.14	14.8	0.03	1.1	19.7
MC-D4	3754	0.20	66.8	271.5	0.18	19.8	0.6	2.7	41.2
MC-F6	458	0.06	63.0	18.1	0.11	19.8	0.1	0.9	7.3
MC-K10	340	3.49	37.5	22.5	0.07	12.0	3.4	0.7	5.3
IC-#1	404	0.59	54.8	40.0	0.18	10.8	6.1	1.2	16.3
IC-#6	405	7.5	30.5	38.6	0.12	15.0	8.2	2.0	14.2
IC-#7	422	11.8	47.3	29.6	0.18	19.5	22.0	5.4	17.3
IC-#8	255	2.88	49.2	17.3	0.11	11.2	7.2	1.9	11.2

Gravity Characterization

Single-pass tests on 2 kg batches, were run at 74 µm through a Falcon concentrator at 200G which removed denser and coarser particles into an average of 3.84% of the total mass. Hand panning the concentrates reduced the mass pull to 0.133% yielding average grades in the order of 800 g/t Au, 60 g/t Ag, 52.4% Fe, 2.84% As, with some upgrading of base metals and depletion of the gangue to <1% Al, Ca, Mg. About 90% of >8 g/t Au in pan tails, and 67.5% in >1.1 g/t primary Falcon tails appeared to be cyanide-soluble (Table 13.14, below).

The results indicate incomplete removal of refractory gold portions by gravity scalping so a series of diagnostic leach tests were conducted on the least extractable interval samples: IC-#6, IC-#7, MC-B2 and MC-D4. High Cr, Co, and Ni levels accompanying iron in the pan concentrates, indicate that varying amounts of grinding scat could also be diluting the gravity concentrate grades.

Diagnostic Leach Results

Selected samples were pulverized and tested by staged sequences of cyanide leaching. Findings are shown in Table 13.14, these results are also presented in Figure 13.1.

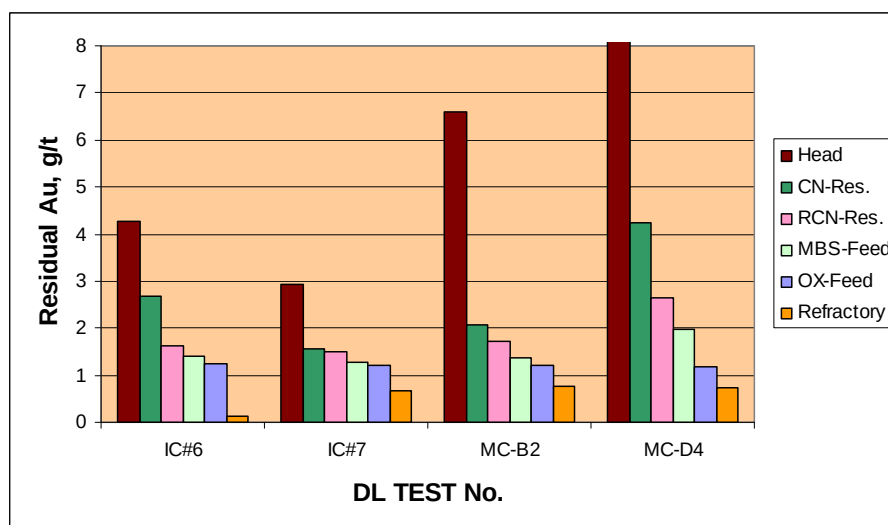
The calculated head grade of MC-D4 (21.3 g/t) mainly represents free milling gold (80% of total) that would leave a residue grade of 4.2 g/t in 24 h using cyanide-leaching. Leaching with Malonitrile CH₂(CN)₂ would yield a grade of 2.7 g/t by stripping Au from preg-robbers. Leaching with acids in neutral or redox environments then releases small portions of Au that are CIL extracted.

The other samples were similar in nature whilst containing lesser proportions of free milling gold. The final oxidative leach step released much of the residual Au, especially from sample IC-#6, suggesting general encapsulation in sulphides. However, it was decided to postpone additional diagnostic testing, including mineralogy on the gravity products, as the potential advantages of producing a gravity concentrate were perceived to be outweighed by other factors. Efforts were then concentrated on systematic leach optimization studies.

Table 13.14
Summary of Diagnostic Leach Findings

Test & Interval ID	DL1 on IC #6 Au		DL2 on IC #7 Au		DL3 on MC-B2 Au		DL4 on MC-D4 Au	
Leach Stages & Cumulative Results	g/t	% Rec	g/t	% Rec	g/t	%Rec	g/t	%Rec
Head	4.28	-	2.92	-	6.60	-	21.25	-
Res.1 Cyanide Soluble	2.68	37.45	1.56	46.59	2.09	68.40	4.24	80.04
Res.2 Carbonaceous minerals	1.62	62.22	1.48	49.21	1.73	73.76	2.65	87.51
Res.3 Calcite/dolomite/pyrrhotite minerals	1.39	67.51	1.28	56.24	1.36	79.35	1.97	90.75
Res.4 Refractory oxide minerals (spinel)	1.24	70.95	1.21	58.60	1.20	81.79	1.19	94.40
Res.5 Sulphides (Py, AsPy and Marcasite)	0.12	97.13	0.68	76.70	0.78	88.19	0.73	96.54

Figure 13.1
Diagnostic Leach Residue Grades



Head grade for MC-D4 is 21.25 g/t Au which is higher than graph boundaries.

13.3.6.3 Whole Ore Leach Optimization

Testing was conducted in several rounds with variations of major parameters of interest, starting with the primary grind size. As reflected by the initial results on gravity-scaled feed, high NaCN maintenance levels and 72 h retention times were abandoned, once it had been established that they could be mitigated. As a result, the reliability of reagent consumptions reported from early phases of testing is considered to be low.

Grind Sensitivity Results

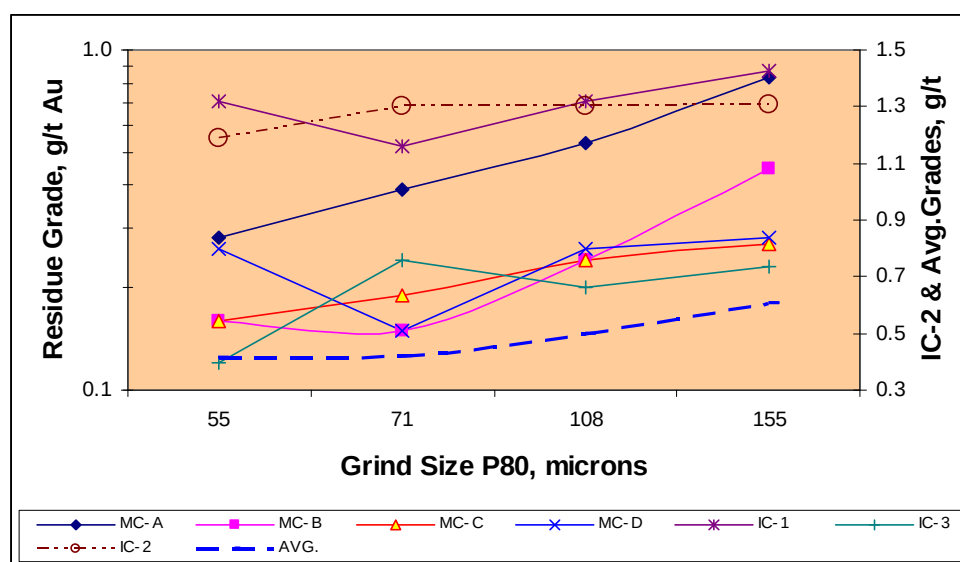
Persistent scatter in calculated head grades affect the Au-recovery levels when the majority of the gold is free-milling as in the Rusoro samples. From the start, averaged residual grades were preferred for trend evaluations, whilst weighted averages may be calculated when major tonnages and blending formulas have been verified. This procedure, as applied in Table 13.15, simplifies interpretations. Plotting of the residue grades (see Figure 13.2) shows some

sensitivity to the grind size, which suggests that liberation and matrix permeability is a concern for most of the ore types tested by baseline direct cyanidation.

Table 13.15
Baseline Grind-Sensitivity Data for Leach Evaluation

Composite ID	Tests C8-C14		Tests C1-C7		Tests C15-C21		Tests C22-C28	
	P ₈₀ µm	g/t Au	P ₈₀ µm	g/t Au	P ₈₀ µm	g/t Au	P ₈₀ µm	g/t Au
MC-A	51.7	0.28	73.1	0.39	114.4	0.53	166.5	0.83
MC-B	52.2	0.16	66.6	0.15	105.1	0.24	167.0	0.45
MC-C	52.0	0.16	68.3	0.19	110.3	0.24	164.5	0.27
MC-D	56.6	0.26	64.5	0.15	104.2	0.26	166.0	0.28
IC-1	56.2	0.71	74.6	0.52	111.8	0.71	146.8	0.87
IC-2	57.0	1.19	75.6	1.30	99.5	1.30	135.6	1.31
IC-3	56.7	0.12	74.3	0.24	111.0	0.20	141.2	0.23
AVG	54.6	0.41	71.0	0.42	108.0	0.50	155.0	0.61

Figure 13.2
Grind Sensitivity of Whole Ore Leaching



Subtle grade variations in residues are magnified by a logarithmic axis and show a mild ambiguity due to heterogeneities and proximity to detection limits. Main trends and un-weighted averages, however, show that grinding to a leach feed F₈₀ of 55 and 74 µm was similar, with a clear recovery-drop for the coarser grinds.

Oxidative Leach Results

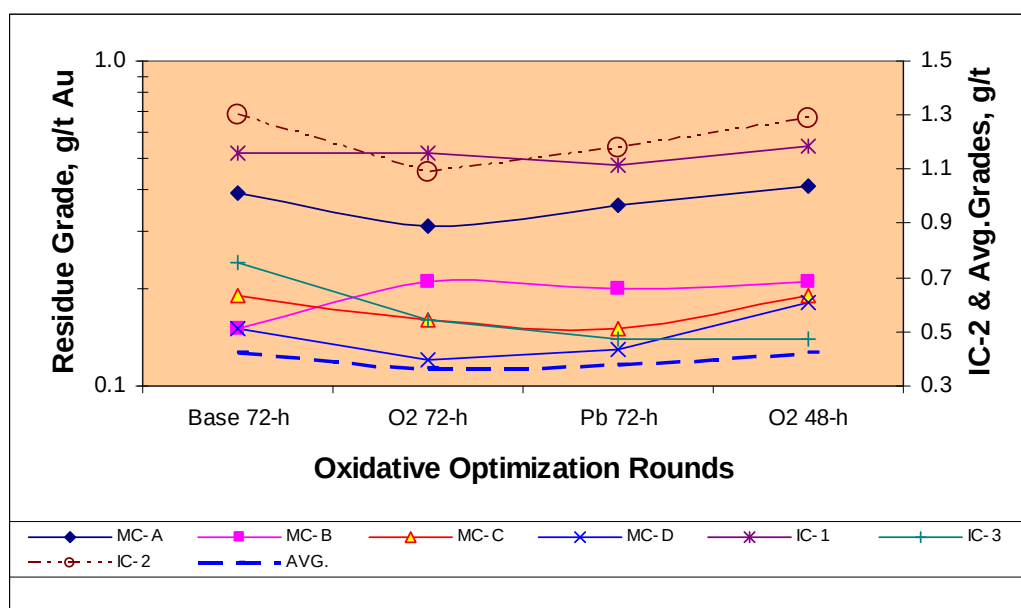
Additions of 800 g/t lead nitrate were compared to the use of oxygen in whole ore cyanidation at various NaCN levels (Table 13.16, Figure 13.3). The results suggest that the use of oxygen is equivalent to the addition of lead nitrate in achieving lower residue grades as compared to the baseline leach, except for MC-B. This allowed a reduction in retention time to 48 h, with maintenance of NaCN at low levels (<0.2 g/L). Calculated Au-recoveries

reflect the trends established by the residue grades, and would average ~90% if the IC-2 sample is discounted.

Table 13.16
Effect of Oxidative Leach Enhancements at 74 μ m

Composite	Tests C1-C7		72-h, O ₂ C29-35		72-h-Pb-nitr.C36-42		48-h O ₂ C50-56	
ID	Au % Rec	g/t Au	Au %Rec	g/t Au	Au % Rec	g/t Au	Au % Rec	g/t Au
MC-A	92.2	0.39	93.5	0.31	92.6	0.36	89.9	0.41
MC-B	92.7	0.15	92.9	0.21	91.3	0.20	87.6	0.21
MC-C	87.9	0.19	91.6	0.16	92.0	0.15	87.6	0.19
MC-D	89.4	0.15	92.5	0.12	91.7	0.13	85.7	0.18
IC-1	85.7	0.52	88.9	0.52	88.0	0.48	85.9	0.55
IC-2	46.2	1.30	60.0	1.09	57.5	1.18	49.8	1.29
IC-3	93.7	0.24	95.1	0.16	95.7	0.14	93.1	0.14
AVG	84.0	0.42	87.8	0.37	87.0	0.38	82.8	0.42

Figure 13.3
Labile Sulphide Oxidation Effects



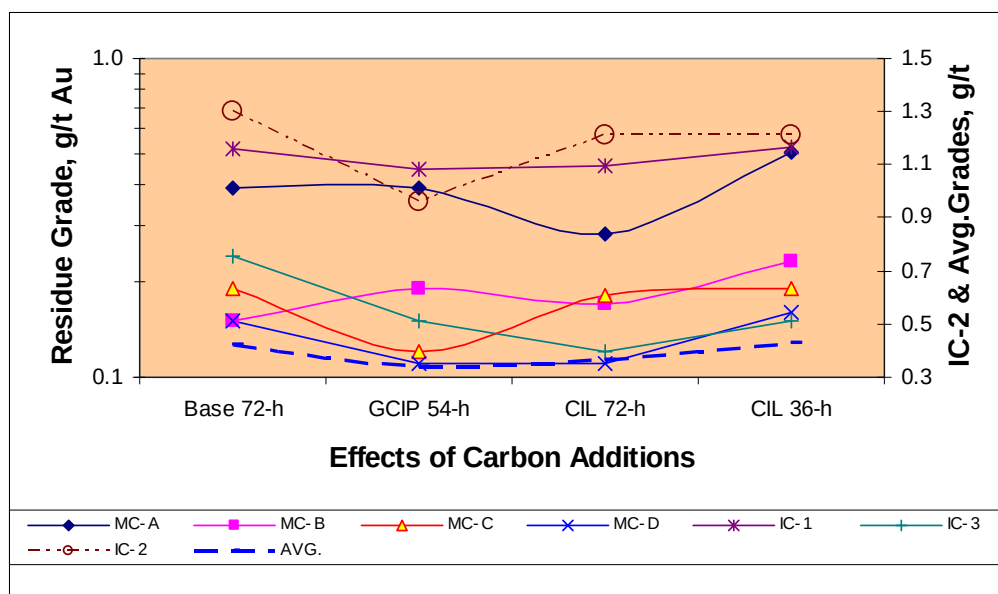
Effect of Carbon Additions

Gravity+CIP test results using conditions 0.5 g/L NaCN and a retention time of 48 + 6 h gave similar results to using direct CIL leaching for 72 h with a higher maintained level of NaCN (0.75 g/L). , as shown in Table 13.17 and Figure 13.4. Halving of the retention to 36-h at 0.2 g/L of maintained NaCN returned the average residue grade to the baseline level of 0.43 g/t Au. Gravity scalping seems to benefit samples MC-C, MC-D, IC-1 and IC-2, but incompletely removed refractory gold portions in all of the samples needed about 48-h of leaching as an optimum.

Table 13.17
Effect of Standard Carbon Additions at 74 μ m

Composite ID	Tests C1-C7		48+6hGCIP9-15		72-h CIL43-49		36-h CIL57-63	
	Au % Rec	Residue (g/t Au)	Au % Rec	Residue (g/t Au)	Au % Rec	Residue (g/t Au)	Au % Rec	Residue (g/t Au)
MC-A	92.2	0.39	90.2	0.39	93.8	0.28	89.4	0.51
MC-B	92.7	0.15	91.7	0.19	93.0	0.17	91.8	0.23
MC-C	87.9	0.19	92.5	0.12	89.9	0.18	87.5	0.19
MC-D	89.4	0.15	91.6	0.11	92.0	0.11	88.5	0.16
IC-1	85.7	0.52	88.6	0.45	85.6	0.46	86.6	0.53
IC-2	46.2	1.30	66.8	0.96	48.9	1.21	60.3	1.21
IC-3	93.7	0.24	94.9	0.15	96.3	0.12	95.2	0.15
AVG	84.0	0.42	88.0	0.34	85.6	0.36	85.6	0.43

Figure 13.4
Effects of Carbon Additions



Whole Ore Leach Overviews

Increased NaCN levels and consumptions had little effect on leach kinetics and recoveries. NaCN consumption also increased continuously with retention time, so that improving kinetics through carbon and oxygen additions, whilst maintaining starvation levels of free cyanide is quite crucial (Figure 13.5). Reduction potentials (RP) were very moderate for sulphide materials, indicating little fouling.

CIP solution data also showed very benign impurity build ups (Table 13.18), with low levels (≤ 15 ppm) of most metals extracted and no signs of cyanide deterioration. Thus, the impurities are likely precipitated or adsorbed on matrix components.

Figure 13.5
Un-weighted Average Performance Summary

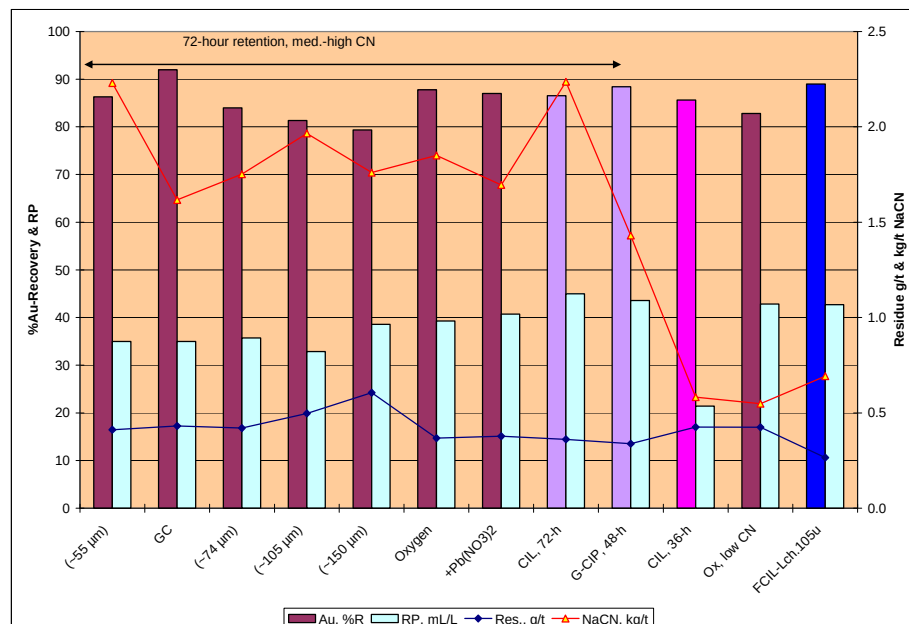


Table 13.18
Typical CIP Solution Assays

Sample Name	Unit	MC-A CIP PLS	MC-B CIP PLS	MC-C CIP PLS	MC-D CIP PLS	IC-1 CIP PLS	IC-2 CIP PLS	IC-3 CIP PLS
Au	mg/l	0.03	0.03	0.02	0.02	0.02	<0.01	0.02
TCN ⁻	mg/l	134.7	125.0	144.7	160.2	143.5	144.6	114.3
WAD CN ⁻	mg/l	127.2	114.5	132.4	134.8	122.0	132.0	104.2
SCN ⁻	mg/l	8.0	8.0	9.0	6.0	12.0	8.0	9.0
CNO ⁻	mg/l	44.0	39.0	42.0	40.0	40.0	40.0	43.0
SO ₄ ²⁻	mg/l	0.4	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Al	mg/l	0.4	<0.2	0.3	0.9	1.2	1.4	0.5
As	mg/l	<0.2	<0.2	<0.2	<0.2	0.7	3.1	0.7
Ca	mg/l	2.0	26.8	3.6	3.8	2.5	2.6	2.0
Cr	mg/l	0.03	<0.01	<0.01	<0.01	0.02	<0.01	<0.01
Co	mg/l	0.05	0.08	0.05	0.03	0.21	<0.01	0.22
Cu	mg/l	4.47	5.51	7.15	2.89	5.01	2.46	4.7
Fe	mg/l	9.19	0.85	1.59	2.85	12.49	12.27	10.93
Na	mg/l	632.0	198.0	285.0	434.0	638.0	603.0	639.0
Zn	mg/l	1.0	0.62	0.62	1.5	0.67	0.73	0.39

A high lime consumption (~3 kg/t) was frequently noted for MC-B and, to a lesser extent (~2 kg/t), also for MC-C (Table 13.19). Residual grades are elevated for IC-2 due to As, and for the higher-grade samples MC-A and IC-1. It appears that incomplete leaching is likely caused by a number of contributing factors, as implied by the diagnostic results. A 48-h retention maintained at 45% solids in 0.2 g/L NaCN, in the presence of carbon, would ensure reliable extractions on the order of 90% Au. Consumptions of 0.6 kg/t NaCN and 1 kg/t lime (to maintain pH ~11) are indicated. The use of oxygen and a finer grind benefits treatment of

the refractory ore types; intensive leaching of gravity concentrate has not been tested, but its overall impact on the whole ore leach is not very pronounced.

Table 13.19
Average Whole Ore Leach Responses per Sample

Composite ID	Au Head Grades (g/t)		Recovery Au (%)	Residual Grade Au (g/t)	Consumption kg/t	
	Measured	Calculated			NaCN	Lime
MC-A	4.62	4.78	90.51	0.45	1.84	0.75
MC-B	1.94	1.80	89.51	0.18	1.62	1.92
MC-C	1.64	1.80	89.51	0.18	1.62	1.92
MC-D	1.19	1.59	88.58	0.17	1.54	0.83
IC-1	3.87	3.98	85.02	0.59	1.57	0.83
IC-2	2.76	2.89	57.52	1.19	1.50	0.65
IC-3	2.71	3.06	94.53	0.16	1.59	0.80
Average	2.68	2.84	85.02	0.42	1.61	1.10

13.3.6.4 Comminution Tests

The results for the selected hard rock composites and Hazen Research Inc. (USA) (Hazen) abrasion indices are shown in Table 13.20 (see Table 13.8 for description of samples).

Table 13.20
Bond Ball Mill Work and Abrasion Indices

Choco 10 Sample Identification	Ball Mill Wi (kWh/t)	Abrasion Index
MET – VBK – 01	12.4	0.18
MET – VBK – 03	12.6	0.14
MET – ROS – 01	16.0	0.67
MET – COA – 01	14.5	0.59

Hazen also determined SMC parameters for SAG mill sizing; results from these tests are shown in Table 13.21

Table 13.21
Summary of SMC Evaluations

Parameters	Value			
	51933-1 MET VBK 01 D41HG	51933-2 MET VBK 03 FWMIN	51933-3 MET ROS 01 MLODE	51933-4 MET COA 01 MLODE
A (maximum breakage)	58.3	67.6	72.1	74.5
b (relation between energy and impact breakage)	0.70	0.44	0.55	0.45
A x b (overall SAG hardness)	40.8	29.7	39.7	33.5
DWi, kWh/m ³	7.1	9.9	6.9	8.4
Mia, kWh/m ³	19.2	24.6	19.7	22.6
ta	0.44	0.31	0.45	0.37
S.G. (Weighing in Air)	2.86	2.93	2.73	2.78

13.3.7 Conclusions and Recommendations

Current plant practice reflects near optimum conditions for whole ore leaching of materials that are largely free-milling in nature. Additions of carbon are needed to shorten retention times at 74 μm grinds, 40% solids, and low levels of NaCN to reduce the reagent consumption.

Systematic studies of whole ore cyanide leaching showed consistent trends in materials from many zones that are to be processed in the Choco mill. Major parameters to be considered in the design of an expansion are:

- Material types and head grade variations, which could be tracked routinely.
- Medium grind requirements with many practical implications.
- Leach kinetics and reagent consumptions, including carbon and lime additions.
- Material blending and sorting routines as related to parametric stability.

In general, 48-h CIL Au extractions of $\geq 90\%$ can be expected at $\sim 74 \mu\text{m}$, 40% solids, ≤ 0.5 g/L NaCN and pH ~ 11 , with typical consumptions of 0.6 kg/t NaCN and < 1 kg/t lime. The majority of leach residues graded ~ 0.2 g/t for Choco-10, but refractory components would give rise to excursions above this value. Careful metallurgical control due to head grade variability and different leach train processing will be required to enable optimization of gold recovery.

Tracking of CN-soluble gold for resource modelling purposes is recommended, as a basis for blending a more predictable feed.

Current plant practice offers a reliable basis for project evaluation as well as a benchmark for further improvements. The current study has focused on a limited portion of the materials to be treated only. Master composites could be selected to test further process optimization, followed by variability testing of the deposit.

13.4 AMMTEC (AUSTRALIA) TESTING

13.4.1 Comminution and Confirmatory Leach Testwork

13.4.1.1 Sample Information

A total of six ore samples from the deposit comprising $\frac{1}{2}$ and $\frac{1}{4}$ drill core samples and one run of mine rock pieces, were shipped to Ammtec for determination of grind, rheology, and thickener design parameters and confirmatory testing of the previously determined CIL conditions. Table 13.22 lists the samples and assay details. The test samples description and source locations are shown in Table 13.23.

Table 13.22
Sample Identification

Sample Location & ID	Au ₁ (g/t)	Au ₂ (g/t)	Ag (g/t)	S _{Total} (%)	S _{Sulphide} (%)	SG
Elisa Zone 09 – 1	7.16	7.06	0.4	1.42	1.35	2.90
Culebra Zone 09 – 2	2.63	2.51	<0.3	0.43	0.31	2.73
Christina Zone 09 – 03	1.61	1.75	<0.3	0.54	0.45	2.89
VBK High Grade 09 – A	14.5	14.6	0.3	1.82	1.82	2.91
VBK Footwall 09 – B	3.78	3.01	0.3	1.38	1.37	2.91
Rosika Pit 09 – C	1.56	1.57	0.4	3.67	3.70	2.87
Coacia 09 – D	1.96	1.66	<0.3	0.82	0.79	2.80

Table 13.23
Sample Description and Source Location

Ore Source	Sample Identity	Description	Mass (kg)
Choco 10	09 – A	VBK High Grade (CH06-634 / CH05-289/CH06-636)	45
	09 – B	VBK Footwall (CH04-222 / CH06-625)	44
	09 – C	Sample from Active Rosika Pit (ROM rock pieces)	85
	09 - D	Coacia (CH-396 / H04-140)	65
Increible 6	09 – 1	Elisa Zone (IC07-329 / IC-07-428)	40
	09 – 2	Culebra Zone (IC-07-355 / IC-07-389)	36
	09 – 3	Christina Zone (IC-07-376 / IC-04-437 / IC07-444 / IC-07-448)	32

13.4.1.2 JK Drop Weight Testwork

The JK Drop Weight test provides ore-specific parameters for use in the JKSimMet Mineral Processing Simulator Software. The parameters are combined with equipment details to analyse and/or predict SAG/autogenous mill performance. The results from the JK Drop Weight test are shown in Table 13.24. Mill sizing requirements can also be derived from other rock testing parameters such as the SMC test, and standard Bond rod and ball mill work indices. Since there are a number of mill sizing techniques available the JK test was only carried out on one sample to supplement the information obtained from the other tests mentioned above.

Table 13.24
JK Tech Drop-Weight Test

Sample Identity	AG/SAG Mill Parameters			
	A	b	A x b	t _a
09-C: ROM Sample from Active Rosika Pit	54.0	0.71	38.3	0.22

The SMC tests were carried out on all the samples in addition to Bond rod mill Bond work index determinations. Ball mill work indices previously determined by PRA in Canada are reported in Table 13.20.

13.4.1.3 SAG Mill Comminution Testwork

SAG Mill Comminution (SMC) testwork was carried out on the fresh samples to re-confirm the parameters for mill sizing as initial work carried out in 2009 had contained samples which were possibly not representative of the future ore. The values derived agreed reasonably well those previously obtained, indicating that the future ore would be hard and that pebble crushing would be required. Table 13.25 lists the results.

Table 13.25
SMC Test Results

Sample Identity	DWi	SG	Derived Values					
			A	b	A x b	Category	Rank	%
09-1: Elisa Zone	6.64	2.89	61.0	0.71	43.3	Medium	1307	43.4
09-2: Culebra zone	8.33	2.70	71.6	0.45	32.2	Hard	545	18.1
09-3: Christina Zone	8.00	2.84	60.1	0.59	35.5	Hard	771	25.6
09-A: VBK High Grade	8.45	2.86	67.3	0.50	33.7	Hard	635	21.1
09-B: VBK Footwall	7.86	2.85	55.4	0.65	36.0	Hard	812	26.9
09-C: Rosika Pit	7.63	2.82	77.1	0.48	37.0	Hard	889	29.5
09-D: Coacia	10.05	2.77	76.4	0.36	27.5	Very Hard	253	8.4

13.4.1.4 Bond Rod Mill Testwork

Bond rod mill work index tests had not previously been conducted and it was felt desirable to do these as they also help indicate a requirement for pebble crushing in the SAG mill circuit. The results (Table 13.26) were significantly higher than those obtained from the ball mill work index tests, which is a good indication that the ore is very competent and hard, and that a pebble crushing installation will be required.

Table 13.26
Bond Rod Mill Work Index Determinations

Sample Identity	Micrometres		Grp (g/rev)	Test Aperture Pi (µm)	Bond Rod Mill Work Index (kWh/t)
	F ₈₀	P ₈₀			
09-1: Elisa Zone	10616	842	6.135	1180	17.5
09-2: Culebra zone	10661	832	3.988		22.6
09-3: Christina Zone	10424	815	4.492		20.8
09-A: VBK High Grade	10470	768	4.781	1180	19.2
09-B: VBK Footwall	10475	768	4.681		19.4
09-C: Rosika Pit	11199	795	4.581		19.9
09-D: Coacia	10798	802	3.773		22.8

13.4.1.5 Bond Abrasion Index Determinations

Bond abrasion index tests results (Table 13.27) demonstrate considerable variation from mildly to very abrasive with a peak value of 0.54 for the Culebra zone. Fortunately this portion of the resource is fairly small so its impact will be limited.

Table 13.27
Bond Abrasion Index Determinations

Sample Identity	Feed Particle Size (mm)	Bond Abrasion Index (Ai)
09-1: Elisa Zone	-19.0 + 12.7	0.25
09-2: Culebra zone		0.54
09-3: Christina Zone		0.22
09-A: VBK High Grade	-19.0 + 12.7	0.37
09-B: VBK Footwall		0.1
09-C: Rosika Pit		0.38
09-D: Coacia		0.47

13.4.1.6 Extraction and Ancillary Testwork

Carbon-in-Leach (CIL) Cyanidation Time Leach

A wide variety of test conditions were employed during the testing conducted in Canada to achieve optimum leach recoveries. To confirm the results, tests were repeated on the fresh samples that had been shipped to Ammtec. The conditions and results are listed in Table 13.28.

Table 13.28
Carbon-in Leach and Cyanidation Leach Tests

Sample Identity	Test No	Grind Size P ₈₀ (µm)	% Gold Extraction After (Hours)						Consumption (kg/t)	
			2	4	8	24	48	72	Lime	NaCN
09-1 Elisa Zone	HS22528	75	69.95	74.41	81.46	85.92	87.08	87.77	0.29	0.75
09-2 Culebra zone	HS22529		45.11	47.37	49.62	51.91	53.06	53.39	0.26	0.78
09-3 Christina Zone	HS22530		74.66	80.53	86.93	91.36	93.07	94.13	0.34	0.86
09-A VBK High Grade	HS22524	75	74.63	85.53	88.56	92.36	92.99	93.41	0.28	0.83
09-B: VBK Footwall	HS22525		62.04	81.17	84.02	87.24	89.25	90.18	0.33	0.85
09-C Rosika Pit	HS22526		65.59	77.89	81.31	82.99	85.16	86.33	0.26	0.55
09-D Coacia	HS22527		69.72	74.46	80.70	87.20	89.82	91.22	0.23	0.67

Rheology Testwork: Viscosity vs Pulp Density and pH

To assist in and confirm slurry pump design, rheology tests were conducted on a sample representing the largest proportion of material to be treated in the initial years of operation. The results (Table 13.29) indicate that the slurry characteristics are typical for material at the

expected slurry density and grind size. Therefore, design requirements are considered standard for this material.

Table 13.29
Rheology Testwork

09 – B VBK FOOTWALL GRIND SIZE P ₈₀ 75µm						
Shear Rate (sec ⁻¹)	Slurry Viscosity (cps)					
	Pulp Density 45% w/w		Pulp Density 55% w/w		Pulp Density 65% w/w	
	pH Natural	pH 10.5	pH Natural	pH 10.5	pH Natural	pH 10.5
4.2	Low	Low	Low	Low	Low	2291
7.4	Low	Low	Low	Low	Low	1484
13.1	Low	Low	Low	Low	Low	997
21.9	Low	Low	Low	Low	91	697
38.9	Low	Low	Low	67	74	478
67.4	27	33	39	67	80	326
119.2	41	45	55	75	92	263
209.5	60	67	73	102	114	291

Flocculation and Thickener Design Settling Testwork

Settling testwork was conducted on a portion of the sample taken from the VBK pit footwall. It was considered that this material was a good representation of future material and would form a basis to confirm preliminary design calculations. The results (Table 13.30) indicate that the proposed design flux is slightly conservative. However, to allow for the possibility of finding additional saprolitic material, which has poor settling characteristics, the thickener sizing was unchanged.

Table 13.30
Thickener Design Testwork

Run No.	Diluted Feed			Flocculant		Underflow		O'flow Clarity (mg/l)	Thickener Size Diameter (m)
	Flux (t/m ² h)	Liquor RR (m/h)	Calc. Solids (%w/w)	Type	Dose (g/t)	Meas. Solids (%w/w)	YS (Pa)		
1	1.00	3.91	22.0	MF10	20	62.3	12	<100	35
2	0.50	1.96	22.0	MF10	20	65.4	21	<100	49
3	1.50	5.86	22.0	MF10	20	60.2	11	<100	29
4	1.50	5.86	22.0	MF10	15	60.4	11	<100	29
5	1.50	5.86	22.0	MF10	10	59.7	10	<100	29

13.5 TESTWORK CONCLUSIONS

The 2008 testwork at the Inspectorate America Corporation, PRA Metallurgical Division laboratories and that conducted by Ammtec, Australia support the results as reported in the Preliminary Assessment (Buchanan et al., 2009). Resultant minor changes to the original flowsheet are:

- The overall leach time has been reduced and the carbon in leach time increased, as it is believed that “pregnant solution robbing” may be occurring. Overall leach/CIL time will remain at approximately 48 h.

- The inclusion of oxygen injection will enhance gold dissolution rates.

In Micon's opinion, the work has confirmed that the material is amenable to SAG milling, that the flowsheet being considered is appropriate for the proposed plant expansion, and that a gold recovery of 89% from a mill head grade of 2.5 g/t is achievable.

14.0 MINERAL RESOURCE ESTIMATES

14.1 CHOCO 10

14.1.1 Previous Estimate

A previous mineral resource estimate by Gold Fields was audited by Micon in 2007 and in 2009 that estimate was used in a preliminary assessment of the Choco 10 deposit (Buchanan et al, 2009). The Gold Fields estimate (Table 14.1) considered all relevant geological data available at that time. The 3D block model used a kriged uniform conditioning methodology which was very selective and complex in nature.

Table 14.1
Gold Fields Mineral Resource Estimates as of September 30, 2007

Deposit	COG (g/t)	Measured			Indicated			Total M + I			Inferred		
		(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Rosika	0.5	1.9	3.17	193	11.8	2.19	829	13.7	2.32	1,021	6.1	2.12	415
Coacia	0.5	0.1	1.18	4	6.3	2.11	428	6.4	2.10	432	3.8	1.72	210
Pisolita	0.5	0.6	2.22	43	2.5	1.52	122	3.1	1.67	166	7.1	2.32	529
VBK	0.5	0.0	0.00	0	35.6	2.66	3,047	35.6	2.66	3,047	25.9	2.24	1,861
TOTAL	0.5	2.6	2.87	240	56.2	2.45	4,426	58.8	2.47	4,665	42.9	2.19	3,016

During its ownership of the property, a series of procedures was developed by Gold Fields to act as guides to all of the geological operations. These resource procedures are documented in MRM_P021 (Resource Modelling - Geological Interpretation), MRM_P022 (Resource Modelling - Geostatistical Analysis) and MRM_P023 (Resource Modelling - Estimation and Classification).

Each drill hole was meticulously logged and examined in great detail. That detail was transferred to the geological models of the deposit. Each mineralized vein system or “domain” (Gold Field terminology) within each deposit was separated, geostatistically analysed and estimated. Each block within the 3D block model had 43 separate attributes. Extracting useful information to guide future exploration has subsequently proved difficult and time consuming. Experience has also shown that the Gold Fields ‘domains’ are too narrow to be of use in mine planning and grade control. Rusoro has also found that differences between domains are too subtle, since they are sufficiently similar to one another that treating each domain separately is unnecessary. Therefore, Rusoro engaged Micon to provide a simpler model that could better meet the demands of an active mining operation.

At the same time, there has been a substantial amount of additional drilling completed within the Choco 10 concession since the last resource estimate. Although some of the drilling has been condemnatory or geotechnical in nature, the majority has been for resource definition and delineation. This has also led to the need to update the mineral resource estimates of these deposits.

14.1.2 Methodology

The new resource model has been developed using Gemcom Software International Inc.'s Surpac Software, version 6.1.3. Surpac is a 3D geological and mining software package that can accurately manipulate and portray geological data and mine development and production headings.

A detailed description of the block model and interpolation can be found in the Choco 10 Mineral Resource Technical Report (Makepeace, 2010); however, the major stages in building the current model are outlined below.

14.1.2.1 Topography

The topography was derived from previous survey data in UTM coordinates. The information was contoured and triangulated to create a digital terrain model (DTM). In Surpac software, a DTM is a combination of a string file (".str") and a triangulation file (".dtm"). A separate DTM was created from the Choco 10 surveyed pit limits as of December 31, 2009. The two DTMs were amalgamated into one seamless DTM

14.1.2.2 Drill Data

The drill data was derived from a series of Excel spreadsheets generated by Rusoro's Maxwell Data Model database (i.e. Collar_PMG.xls, Survey_PMG.xls, Assay_PMG.xls, Lithology_PMG.xls, Stratigrafia_PMG.xls).

These files were checked and converted to input files for the Surpac drill database (Microsoft Corporation's Access database engine). The resulting drill database file was created (i.e. rcpvbk_uc.ddb).

14.1.2.3 Deposit Models

The known deposits in the Choco 10 concession [Rosika (R), Coacia (C), Pisolita (P) and Villa Balazo-Karolina (VBK)] are closely associated with one another. There are three dimensional overlaps between each of the deposits. Although originally considered two separate deposits, the Rosika and Coacia deposits are in essence one continuous deposit. The deposits are also mined in one long open pit. To compare the current mineral resources with the Gold Fields mineral resources, it was necessary to construct a wireframe constraint to separate the Rosika and Coacia deposits.

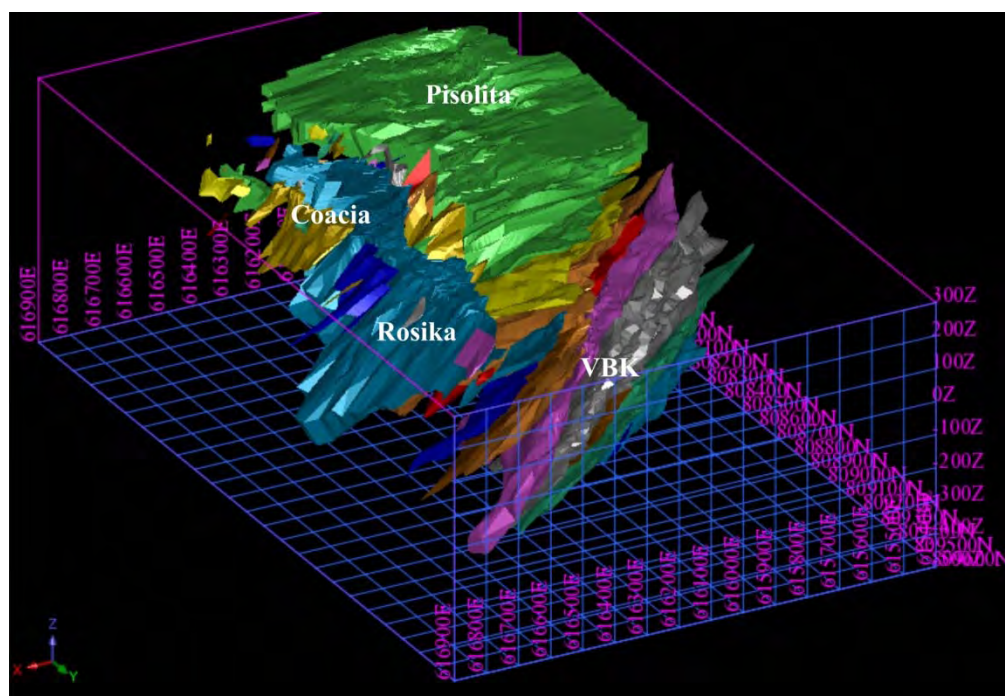
14.1.2.4 Geological Wireframe Models

Three dimensional geological wireframe models constrain the search for composited assays during the resource estimation. The polylines within the wireframe models must be closed and have no cross-overs, overlaps, spikes and duplicate points before a triangulation algorithm can be used and a valid solid can be created.

A new set of wireframes were created by generating 20 metre sections through each of the deposits based on the Gold Fields wireframes and the drill database. The stacked vein systems were digitized as larger units that incorporated mineralized haloes and their associated low-grade and high-grade lenses. The resulting polylines were triangulated into solids that became separate entities and validated.

The geological wireframe models for all four Choco 10 deposits is illustrated in Figure 14.1.

Figure 14.1
VBK+RCP Geological Wireframe Model
(Looking Southwest)



14.1.2.5 Compositing

The drill database samples were taken over varying interval lengths with respect to either geology and/or visible mineralization. The minimum interval was 0.01 m while the maximum interval was 6.10 m with an average of 1.18 m. The total number of assays was 264,046.

A 2 m fixed length downhole composite was chosen with a minimum 75% of the sample interval being included. Remnants are included in a second string range number (i.e., 2). A global composite was created. This was further constrained to the four mineralized wireframes. Table 14.2 is a summary of the constrained database in the four deposits which is illustrated in Figure 14.2 and Figure 14.3.

Table 14.2
Deposit Basic Statistics

	Rosika	Coacia	Pisolita	VBK
Number of samples	8,683	5,601	19,709	25,610
Maximum Value	81.70	37.59	35.00	83.77
Mean	1.159	0.917	0.361	0.877
Variance	7.378	5.889	1.054	6.886
Standard Deviation	2.716	2.427	1.027	2.624
Coefficient of Variation	2.344	2.646	2.841	2.993
90 th Percentile	3.105	2.196	0.800	2.141
95 th Percentile	5.080	4.112	1.473	4.115
99 th Percentile	11.443	12.386	4.238	12.281

Figure 14.2
Rosika and Coacia Histogram and Frequency Distribution Graphs

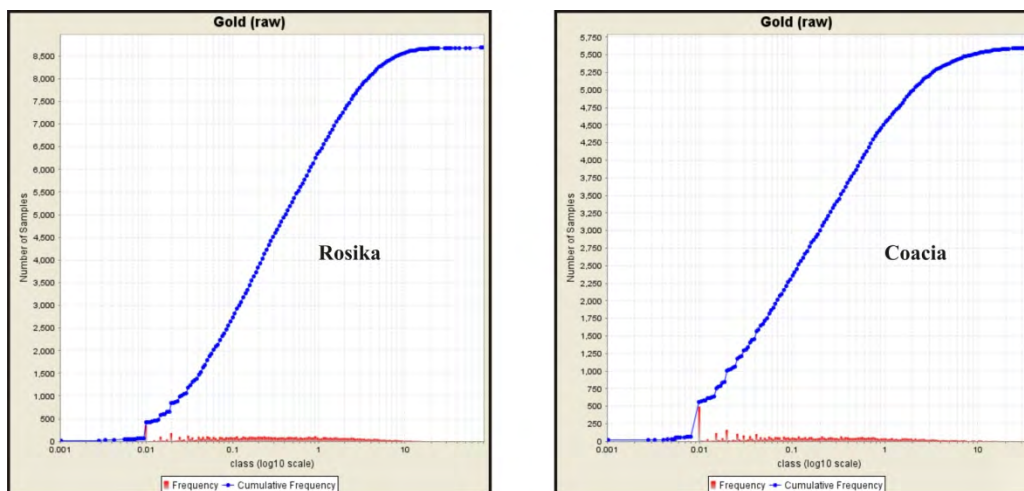
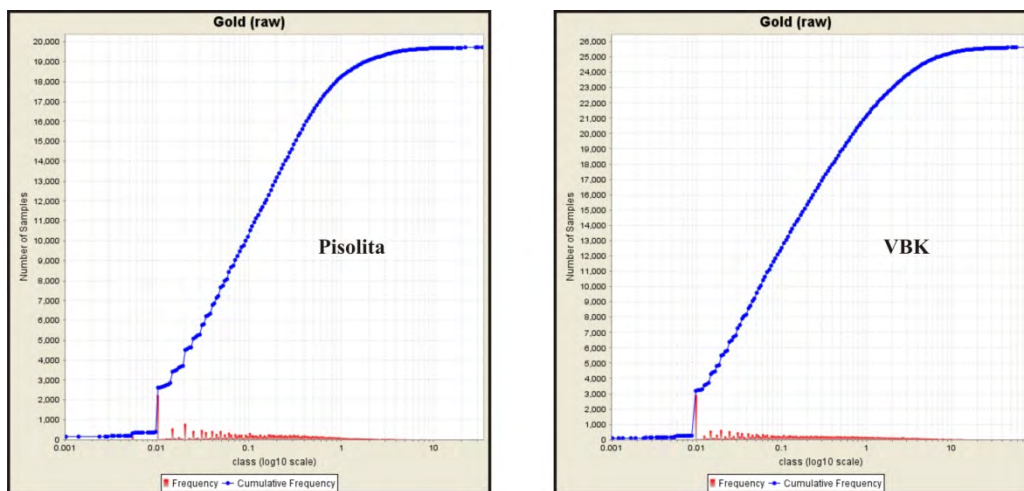


Figure 14.3
Pisolita and VBK Histogram and Frequency Distribution Graphs



The coefficient of variation (i.e., standard deviation/mean) for all four deposits, indicate that outliers exist and that, therefore, capping should be examined.

14.1.2.6 Outliers

The basic statistics of the four deposits seem to indicate a large capping would be required for each deposit. To obtain a coefficient of variation of 1.2 or less, the capping value for the four deposits would range between 1 and 2.5 g/t. The ninety fifth (95%) percentile values range from 0.80 to 3.11 g while the 99% values range from 4.24 to 12.39 g/t. Neither of these two methods produces an acceptable capping value. The mineralization within the wireframes is not strictly high grade mineral lenses. Lower grade mineralization is included which most likely has affected the basic statistics.

An overall cap value of 34.285 g/t Au (1 oz/t) was determined to be adequate for the database. There were only 29 samples out of 59,603 that had to be capped.

14.1.2.7 Variography

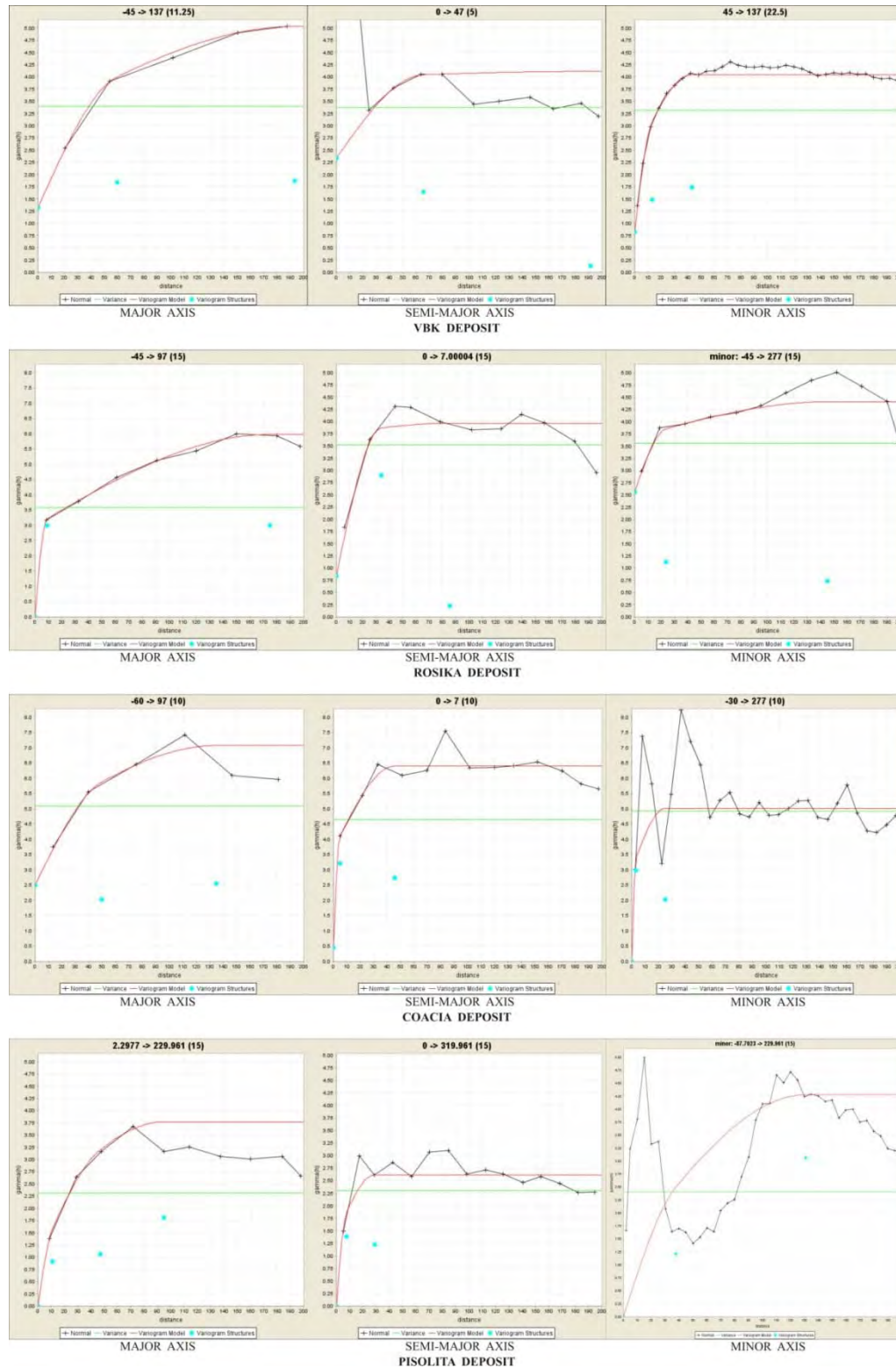
The four composite semi-variograms were calculated for gold. The geological models of the deposit assisted in determining the major, semi-major and minor directions of continuity. The semi-variograms for each of the four known deposits in the Choco 10 concession have been reproduced in Figure 14.4 and the parameters are summarized in Table 14.3.

Table 14.3
Variogram Parameters

Deposit		c0	c1	c2	c3	M:SM	M:m	Plunge	Dip
VBK	Variance	1.320	1.839	1.874	-				
	Range		60	194	-				
	M+I	63 m							
	Bearing	137°						-45°	0°
	Ratio					1.0111	4.4924		
Rosika	Variance	0.000	2.987	2.987	-				
	Range		9	175	-				
	M+I	75 m							
	Bearing	097°						-45°	0°
	Ratio					1.119	2.133		
Coacia	Variance	2.490	2.034	2.544	-				
	Range		50	136	-				
	M+I	43 m							
	Bearing	097°						-60°	0°
	Ratio					1.000	1.397		
Pisolita	Variance	0.000	0.905	1.058	1.807				
	Range		11	47	95				
	M+I	42 m							
	Bearing	230°						+2.3°	0°
	Ratio					1.063	4.582		

Note : M = Major Axis, SM = Semi-major axis and m = Minor axis

Figure 14.4
Semi-Variograms for RCPVBK



14.1.3 Block Model

A block model encompassing both the geological wireframe models of all four deposits as well as the drill hole database was constructed. The block model geometry is documented in Table 14.4.

Table 14.4
Block Model Geometry

	Y	X	Z
Minimum Coordinates	807200	615450	-460
Maximum Coordinates	809500	616850	+280
User Block Size	20	10	5
Minimum Block Size	5	2.5	1.25
Bearing = 0.0°	Dip = 0.0°	Plunge = 0.0°	

The total number of blocks was 6,317,283 with a storage efficiency of 95.85%. Three attributes were created and are listed in Table 14.5.

Table 14.5
Block Model Attributes

Attribute	Type	Decimal	Background	Description
class	character	NA	unestimated	measured, indicated, inferred
density	float	2	-1.00	rock density
gold	float	2	-1.00	grams per metric ton gold

14.1.4 Interpolation Method

The estimation method employed was an Inverse Distance Cubed (ID³) calculation using search ellipsoids based on the variography discussed in Section 14.1.2.7. This method was chosen because the Nearest Neighbour method tends to generate fewer tonnes at a higher grade while the Ordinary Kriging and Inverse Distance Squared methods tend to produce higher tonnes at a lower grade. The Inverse Distance Cubed method tends to produce numbers that fall between these other methods.

The estimates were constrained to individual geological wireframe models acting as hard boundaries. An individual block estimate would not accept composites outside of the wireframes or in adjacent lenses that make up the wireframe model. Rosika and Coacia are constrained further by the Rosika and Coacia deposit model (Section 14.1.2.3).

The block model parameters are documented in Table 14.6.

Table 14.6
Block Model Parameters

Deposit	Rotation (°Az)			Search Radius (m)			Composites		
	Bearing	Plunge	Dip	Major	Semi	Minor	Min	Max	DDH
Measured Resource									
Rosika	097	-45	0	25	22.3	11.7	3	10	2
Coacia	097	-60	0	10	10.0	7.2	3	10	2
Pisolita	230	+2.3	0	14	13.2	3.1	3	10	2
VBK	137	-45	0	20	19.7	4.5	3	10	2
Indicated Resource									
Rosika	097	-45	0	75	67.0	35.2	3	15	2
Coacia	097	-60	0	43	43.0	30.8	3	15	2
Pisolita	230	+2.3	0	42	39.5	9.2	3	15	2
VBK	137	-45	0	63	62.3	14.0	3	15	2
Inferred Resources									
Rosika	097	-45	0	175	156.4	82.0	3	15	6
Coacia	097	-60	0	136	136.0	97.4	3	15	6
Pisolita	230	+2.3	0	95	89.4	20.7	3	15	10
VBK	137	-45	0	193	190.9	43.0	3	15	10

Note : DDH is maximum per drill hole

14.1.5 Block Model Report

The user (maximum) block volume is 20 m (N-S) by 10 m (E-W) by 5 m (vertical). This equates to 1,000 m³. The specific gravity varies within the deposits due to amount of decomposition of the rock. The density is one of the attributes of the block model. The tonnage of a saprolite user block is 1,600 metric tonnes while a saprock user block is 2,350 metric tonnes and fresh rock is 2,700 metric tonnes.

A block model report was generated for each deposit with respect to its resource category and grouped with respect to gold grade cut-offs.

14.1.6 Block Model Validation

The block model and accompanying drill hole database were compared visual in section (north-south and east-west), plan and in 3D. The block model has been constrained to the 3D wireframes. All blocks outside the wireframes have a grade attribute of -1. Visually the blocks and their respective grade attributes correspond well to both grade and 3D location of the mineralized intervals within the database.

The method used (Inverse Distance Cubed or ID³) normally produces tonnage and grade that is between the Nearest Neighbour and the Ordinary Kriging/Inverse Distance Squared (ID²)

methods. Micon believes that the block model results portray the best estimate of the mineralization in the four deposits that make up the Choco 10 property.

14.1.7 Mineral Resources

Mineral resource reporting in Canada follows National Instrument 43-101 and its companion policy 43-101CP and technical report requirements 43-101F1 which have been in-place since February 1, 2001. The mineral resource definitions are based on the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) definitions (CIM Definition Standards – For Mineral Resources and Mineral Reserves, adopted on December 11, 2005).

Under these definitions:

“A Mineral Resource is a concentration or occurrence of natural solid inorganic material or natural solid fossilized organic material including base and precious metals, coal and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.” (CIM, 2005)

There are three subdivisions within the mineral resource category, which are based on decreasing geological confidence (*Measured, Indicated and Inferred*). The Choco 10 deposits have mineral resources in all three categories based on geostatistics. The definitions of the categories are as follows:

Inferred Mineral Resource

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Variable cut-off scenarios were run at 0.1 gram per tonne (g/t) gold intervals. A grade of 0.5 g/t Au was selected as a realistic cut-off value for these mineral resources.

14.1.7.1 Rosika Deposit

At the selected cut-off grade of 0.5 g/t Au, the Rosika deposit contains a measured plus indicated mineral resource of 18.3 million tonnes grading 2.15 g/t Au, and an inferred mineral resource of 12.2 million tonnes grading 1.55 g/t Au.

Table 14.7, Table 14.8 and Table 14.9 document the Rosika mineral resources with variable cut-off scenarios for the three resource categories, respectively.

Table 14.7
Rosika Measured Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	6,389,758	1.06	6,790,901	218,327
0.2	4,897,622	1.34	6,567,081	211,132
0.3	3,993,552	1.59	6,341,064	203,865
0.4	3,471,941	1.77	6,158,500	197,996
0.5	3,078,605	1.94	5,981,499	192,305
0.6	2,779,288	2.09	5,816,874	187,013
0.7	2,504,575	2.25	5,638,311	181,272
0.8	2,249,796	2.42	5,447,226	175,128
0.9	2,006,982	2.61	5,240,835	168,493
1.0	1,821,814	2.78	5,064,925	162,837

Table 14.8
Rosika Indicated Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	32,435,148	1.16	37,610,039	1,209,163
0.2	24,520,346	1.49	36,422,819	1,170,994
0.3	20,319,914	1.74	35,414,715	1,138,583
0.4	17,485,923	1.97	34,422,818	1,106,694
0.5	15,246,466	2.19	33,415,063	1,074,294
0.6	13,729,553	2.37	32,580,760	1,047,471
0.7	12,466,147	2.55	31,759,547	1,021,069
0.8	11,545,163	2.69	31,068,809	998,862
0.9	10,673,804	2.84	30,328,153	975,050
1.0	9,907,523	2.99	29,600,186	951,646

Table 14.9
Rosika Inferred Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	22,533,906	0.95	21,430,056	688,976
0.2	17,832,541	1.16	20,771,865	667,815
0.3	15,338,952	1.32	20,173,404	648,575
0.4	13,829,941	1.42	19,645,250	631,595
0.5	12,165,465	1.55	18,879,591	606,979
0.6	10,308,993	1.73	17,858,532	574,152
0.7	9,408,886	1.84	17,273,462	555,342
0.8	8,695,107	1.93	16,738,128	538,131
0.9	7,999,709	2.02	16,147,039	519,127
1.0	6,768,973	2.21	14,977,840	481,538

14.1.7.2 Coacia Deposit

Table 14.10, Table 14.11 and Table 14.12 document the Coacia deposit mineral resources with variable cut-off scenarios for the three resource categories respectively.

Table 14.10
Coacia Measured Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	328,072	0.94	309,359	9,946
0.2	273,311	1.10	301,145	9,682
0.3	214,449	1.34	286,429	9,209
0.4	186,183	1.49	276,536	8,891
0.5	149,571	1.74	259,695	8,349
0.6	131,394	1.90	249,516	8,022
0.7	115,803	2.07	239,381	7,696
0.8	99,809	2.28	227,386	7,310
0.9	87,836	2.47	217,329	6,987
1.0	77,578	2.68	207,686	6,677

Table 14.11
Coacia Indicated Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	2,755,054	0.79	2,188,260	70,353
0.2	2,092,876	1.00	2,088,933	67,159
0.3	1,677,538	1.18	1,985,099	63,821
0.4	1,357,329	1.38	1,873,026	60,218
0.5	1,175,906	1.52	1,791,385	57,593
0.6	1,005,993	1.69	1,697,933	54,589
0.7	894,130	1.82	1,625,222	52,251
0.8	773,827	1.98	1,534,995	49,350
0.9	676,193	2.15	1,452,006	46,682
1.0	603,741	2.29	1,383,177	44,469

Table 14.12
Coacia Inferred Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	19,234,061	0.87	16,791,969	539,862
0.2	15,212,368	1.06	16,188,715	520,467
0.3	12,052,045	1.28	15,430,237	496,082
0.4	10,000,442	1.47	14,712,176	472,996
0.5	8,329,574	1.68	13,960,286	448,823
0.6	7,157,854	1.86	13,327,557	428,481
0.7	6,231,896	2.04	12,725,684	409,131
0.8	5,426,812	2.23	12,121,871	389,718
0.9	4,689,164	2.45	11,494,870	369,560
1.0	4,040,849	2.69	10,878,971	349,759

At the selected cut-off grade of 0.5 g/t Au, the Coacia deposit contains a measured plus indicated mineral resource of 1.3 million tonnes grading 1.55 g/t Au, and an inferred mineral resource of 8.3 million tonnes grading 1.68 g/t Au.

14.1.7.3 Pisolita Deposit

At the selected cut-off grade of 0.5 g/t Au, the Pisolita deposit contains a measured plus indicated mineral resource of 5.4 million tonnes grading 1.58 g/t Au, and an inferred mineral resource of 18.7 million tonnes grading 1.15 g/t Au.

Table 14.13, Table 14.14 and Table 14.15 document the Pisolita deposit mineral resources with variable cut-off scenarios for the three resource categories respectively.

Table 14.13
Pisolita Measured Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	473,079	0.86	405,285	13,030
0.2	360,234	1.08	389,486	12,522
0.3	308,430	1.22	377,053	12,122
0.4	249,184	1.43	356,317	11,456
0.5	206,227	1.63	336,987	10,834
0.6	183,332	1.77	324,852	10,444
0.7	171,912	1.85	317,429	10,205
0.8	143,256	2.06	295,651	9,505
0.9	129,198	2.19	283,561	9,116
1.0	118,217	2.31	273,129	8,781

Table 14.14
Pisolita Indicated Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	14,255,108	0.73	10,381,603	333,769
0.2	10,260,667	0.95	9,782,436	314,505
0.3	7,996,133	1.15	9,216,303	296,304
0.4	6,414,400	1.35	8,662,696	278,506
0.5	5,143,825	1.57	8,090,938	260,124
0.6	4,310,605	1.77	7,632,667	245,390
0.7	3,686,924	1.96	7,227,274	232,357
0.8	3,215,684	2.14	6,873,844	220,994
0.9	2,813,196	2.32	6,531,729	209,995
1.0	2,493,161	2.50	6,227,696	200,220

Table 14.15
Pisolita Inferred Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	65,608,430	0.50	33,059,776	1,062,872
0.2	45,244,392	0.66	30,005,170	964,666
0.3	32,383,913	0.83	26,790,050	861,300
0.4	24,094,329	0.99	23,888,696	768,022
0.5	18,693,635	1.15	21,458,384	689,887
0.6	14,873,153	1.30	19,357,118	622,331
0.7	11,796,292	1.47	17,357,159	558,033
0.8	9,495,954	1.65	15,631,905	502,566
0.9	7,960,388	1.80	14,326,674	460,603
1.0	6,757,719	1.95	13,184,139	423,870

14.1.7.4 VBK Deposit

Table 14.16, Table 14.17 and Table 14.18 document the VBK deposit mineral resources with variable cut-off scenarios for the three resource categories respectively.

Table 14.16
VBK Measured Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	62,722,208	1.22	76,678,359	2,465,209
0.2	49,236,364	1.52	74,790,341	2,404,509
0.3	41,332,942	1.76	72,814,485	2,340,986
0.4	35,625,802	1.99	70,816,986	2,276,766
0.5	31,266,890	2.20	68,855,476	2,213,704
0.6	27,865,400	2.40	66,984,656	2,153,557
0.7	25,111,433	2.60	65,194,578	2,096,006
0.8	22,902,818	2.77	63,538,117	2,042,750
0.9	20,855,348	2.96	61,797,767	1,986,798
1.0	19,221,812	3.13	60,245,908	1,936,906

Table 14.17
VBK Indicated Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	185,754,250	0.89	165,860,045	5,332,400
0.2	144,188,893	1.11	159,625,241	5,131,952
0.3	117,789,140	1.30	153,025,303	4,919,763
0.4	98,451,840	1.49	146,257,248	4,702,171
0.5	83,643,195	1.67	139,593,358	4,487,926
0.6	72,628,649	1.84	133,535,357	4,293,162
0.7	64,050,223	2.00	127,959,380	4,113,894
0.8	57,022,100	2.15	122,688,288	3,944,428
0.9	51,293,754	2.30	117,819,194	3,787,887
1.0	46,669,803	2.43	113,426,441	3,646,660

Table 14.18
VBK Inferred Mineral Resource

Grade Cut-Off (g/t Au)	Tonnage (t)	Grade (g/t Au)	Gold	
			(g)	(oz)
0.1	58,049,585	0.75	43,331,166	1,393,097
0.2	42,066,302	0.97	40,933,673	1,316,018
0.3	33,449,125	1.16	38,779,379	1,246,757
0.4	26,076,058	1.39	36,198,806	1,163,792
0.5	19,973,720	1.67	33,452,753	1,075,506
0.6	16,680,268	1.90	31,641,355	1,017,270
0.7	14,244,910	2.11	30,082,726	967,160
0.8	12,663,005	2.28	28,896,297	929,016
0.9	11,419,824	2.44	27,839,593	895,043
1.0	10,530,343	2.56	26,994,586	867,876

At the selected cut-off grade of 0.5 g/t Au, the VBK deposit contains a measured plus indicated mineral resource of 114.9 million tonnes grading 1.81 g/t Au, and an inferred mineral resource of 20.0 million tonnes grading 1.67 g/t Au.

14.1.8 Choco 10 Discussion

Table 14.19 summarises the mineral resource estimates for all four deposits on the Choco 10 property, at a cut-off grade of 0.5 g/t Au. These mineral resources are inclusive of any mineral reserves on the property. Mineral resources that are not mineral reserves do not have demonstrated economic viability. As of the effective date of this report, there are no known legal, political, environmental or other risks that could materially affect the potential development of the mineral resources and reserves.

Table 14.19
Choco 10 Mineral Resource Estimates
Above a cut-off grade of 0.50 g/t Au, as of December 31, 2009

Deposit	Measured			Indicated			Total M + I			Inferred		
	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Rosika	3.1	1.94	192	15.2	2.19	1,074	18.3	2.15	1,267	12.2	1.55	607
Coacia	0.1	1.74	8	1.2	1.52	58	1.3	1.55	66	8.3	1.68	449
Pisolita	0.2	1.63	11	5.1	1.57	260	5.3	1.58	271	18.7	1.15	690
VBK	31.3	2.20	2,214	83.6	1.67	4,488	114.9	1.81	6,702	20.0	1.67	1,076
TOTAL	34.7	2.17	2,425	105.2	1.74	5,880	139.9	1.85	8,305	59.2	1.48	2,821

The current Mineral Resource estimate has delineated a total of 139.9 million tonnes of mineralization with an average grade of 1.85 g/t gold in the Measured and Indicated categories. This translates to an increase of 81.1 million tonnes from the previous Mineral Resource estimate that was completed in 2006/2007. There is also a total Inferred Mineral Resource of 59.2 million tonnes with an average grade of 1.48 g/t gold. It cannot be assumed that all or any part of this Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the Inferred

Mineral Resource is insufficient to allow the meaningful application of technical and economic parameters.

A comparison of the December 31, 2009 mineral resource estimate with the September 30, 2007 (December 31, 2006) mineral resource estimate is presented in Table 14.20.

Table 14.20
Choco 10 Mineral Resource Comparison

Deposit	COG	Measured		Indicated		M+I		Inferred	
		Tonnes (millions)	Au (oz) (000)	Tonnes (millions)	Au (oz) (000)	Tonnes (millions)	Au (oz) (000)	Tonnes (millions)	Au (oz) (000)
Rosika	0.5	+ 1.1	- 1	+ 3.4	+ 245	+ 4.6	+ 245	+ 6.1	+ 192
Coacia	0.5	+ 0.1	+ 5	- 5.1	- 370	- 5.1	- 366	+ 4.5	+ 239
Pisolita	0.5	- 0.4	- 32	+ 2.6	+ 138	+2.3	+ 105	+ 11.6	+ 161
VBK	0.5	+ 31.3	+2,214	+48.0	+1,441	+79.3	+ 3,654	- 5.9	- 786
TOTAL	0.5	+ 32.1	+ 2,185	+49.0	+1,454	+ 81.1	+ 3,639	+ 16.3	- 194

Note: COG = Cut-off Grade

The largest increase is at the VBK deposit. This is due to the 40 strategically placed drill holes within this deposit over the last three years. The drilling has delineated additional tonnage along strike and at depth. It has also upgraded tonnage from the Inferred Mineral Resource category to either Indicated or Measured Mineral Resource categories due in part to reducing the drill spacing within the deposit. A substantial portion of higher grade mineralization appears to be located at depth.

The Rosika deposit Mineral Resources improved, primarily due to its proximity to the VBK deposit. Additional drilling in VBK intersected Rosika mineralization.

The Coacia deposit has also been mined for the last three years which explains the losses to its Mineral Resources. Additional drilling within the boundaries of the deposit assisted in producing modest gains in the Inferred Resource category.

The Pisolita deposit gained some tonnage in the “North Pisolita” area as well as a modest increase in tonnage at depth. This deposit had the most drilling (221 holes) in the last three years which translated into a larger Inferred Resource category.

14.2 INCREIBLE 6

14.2.1 Previous Estimate

A mineral resource estimate was completed in September 2007 using all available data at the time It was constructed by Mr. Zbeetnoff using Gemcom Software Internationals GEMS Software version 5.55. A geological interpretation of the deposit was carried out by Rusoro geological staff which included identifying mineralized lenses as well as the depth to which oxidation has occurred. The interpretation was completed on vertical cross-sections spaced roughly at 25 m. The interpretation was also validated on 20-m level plans as the cross-sectional work advanced. Wireframe solids were constructed from this work. The

interpolation was an ID² method. Assays were capped to reduce the influence of erratic high-grade values. A detailed description of this mineral resource is found in Laudrum et al., 2008. Table 14.21 shows the previous resource estimate for Incredible 6.

Table 14.21
Previous Incredible 6 Mineral Resource Estimate, September 2007

Deposit	Cut Of Grade	Indicated			Inferred		
		(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Culebra	0.5	5.55	1.92	343.1	2.45	1.57	123.5
Christina	0.5	2.58	2.11	174.6	1.93	1.61	99.8
Elisa/Ingrid	0.5	15.08	2.18	1,056.0	11.84	2.11	804.5
Olga/Enoc	0.5	0.24	1.91	14.7	1.32	1.75	74.2
TOTAL	0.5	23.45	2.11	1,588.5	17.54	1.95	1,102.1

Additional drilling and a refinement to the interpretation of the deposits led to a re-evaluation of the mineral resources in 2009-10 as part of this feasibility study.

14.2.2 Methodology

The new model has been developed using Gemcom Software International Inc.'s Surpac Software, version 6.1.3. Surpac is a 3D geological and mining software package that can accurately manipulate and portray geological data and mine development and production headings.

The basic methodology used to generate the Incredible 6 estimate was identical to the methodology used to develop the Choco 10 estimate which was covered in the previous section of this report.

14.2.2.1 Topography

The topography was derived from previous survey data in UTM coordinates. A topography DTM constructed to cover the Ingrid-Elisa, Olga-Enoc and Christina deposits (IOC) and a separate topography DTM was constructed for the Culebra deposit. DTMs were constructed for the saprolite/saprock and saprock/fresh rock contacts for both the IOC and Culebra deposits. These contact DTMs were used to assign the proper specific gravity attributes within the block model.

14.2.2.2 Drill Data

The drill data was derived from an Excel spreadsheet generated by Rusoro's Maxwell Data Model database (i.e. INC6_Micon.xls). The individual worksheets within the spreadsheet included collar, survey and assay data for the entire Incredible 6 concession.

These files were checked and converted to input files for the Surpac drill database (Microsoft Corporation's Access database engine). The resulting drill database file was created (i.e. new_inc6_dh.ddb).

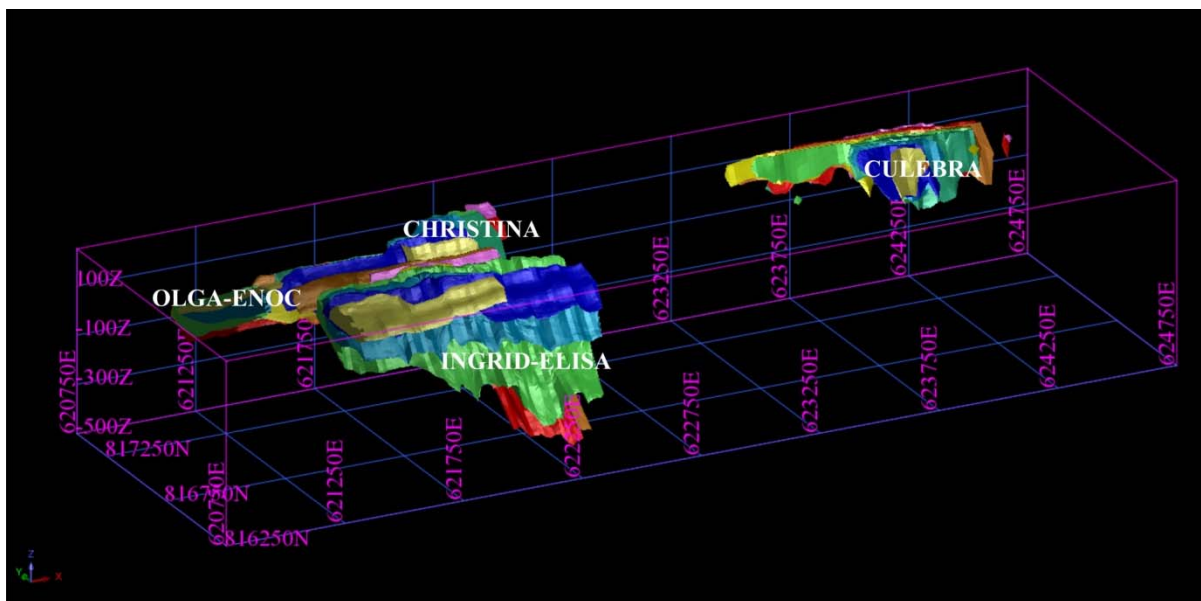
14.2.2.3 Deposit Models

The main deposits in the Incredible 6 concession are Ingrid-Elisa (Ingrid) , Olga-Enoc (Olga), Christina and Culebra. There are other small mineralized zones within the Incredible 6 concession that have been drill tested but at this time they are insignificant compared to the main four deposits.

14.2.2.4 Geological Wireframe Models

A new set of wireframes were created by Rusoro geologist in Datamine Software. The wireframes required re-digitizing in Surpac Software to be validated as solids. The Surpac set of wireframes was essentially a copy of the Datamine wireframes. The resulting wireframes for each deposit is illustrated in Figure 17.5. The individual series of stacked veins (domains) can clearly be seen in Figure 17.5. Each vein (domain) was individually triangulated into a solid and validated.

Figure 14.5
Incredible 6 Deposit Wireframe Solids



14.2.2.5 Compositing

The drill database samples were taken over varying interval lengths with respect to either geology and/or visible mineralization. The minimum interval was 0.01 m while the maximum interval was 67.0 m with an average of 0.97 m. The total number of assays was 119,134.

A 2 metre fixed length downhole composite was chosen with a minimum 75% of the sample interval being included. Remnants are included in a second string range number (i.e., 2). A global composite was created. This was further constrained to the four mineralized wireframes. Table 14.22 is a summary of the basic statistics of the constrained database in the four Incredible 6 deposits and which are illustrated in Figure 14.6 and Figure 14.7.

Table 14.22
Incredible 6 Deposit Basic Statistics

	Ingrid	Christina	Olga	Culebra
Number of samples	7,166	1,358	576	3,016
Maximum Value	173.70	186.73	20.57	68.07
Mean	1.30	0.90	0.69	1.01
Variance	14.63	29.86	2.37	7.17
Standard Deviation	3.83	5.46	1.54	2.68
Coefficient of Variation	2.94	6.08	2.23	2.66
90 th Percentile	3.25	1.98	1.89	2.51
95 th Percentile	5.73	3.46	3.00	4.52
99 th Percentile	14.89	8.59	7.60	10.89

The coefficient of variation (i.e., standard deviation/mean) for all four deposits, indicate that outliers exist and that capping should be examined.

Figure 14.6
Ingrid and Christina Histogram and Frequency Distribution Graphs

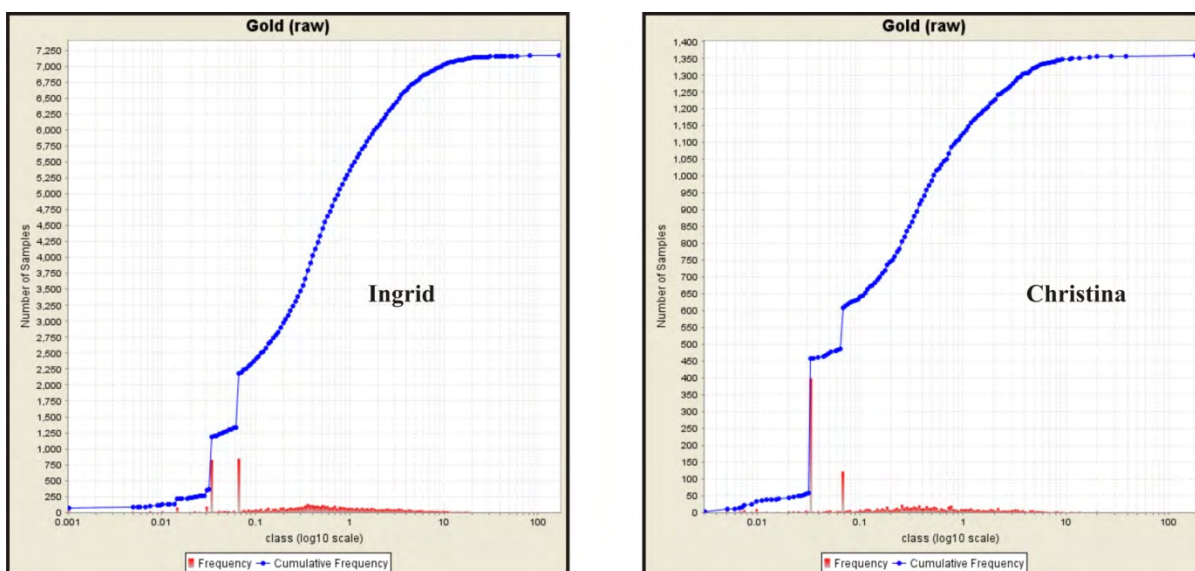
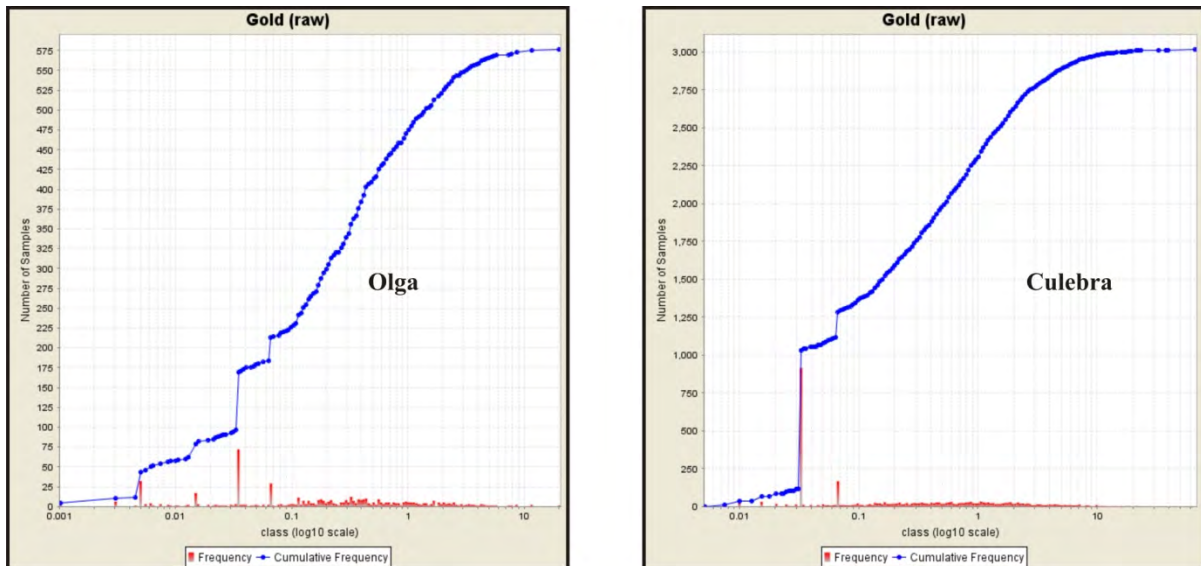


Figure 14.7
Olga and Culebra Histogram and Frequency Distribution Graphs



14.2.2.6 Outliers

The basic statistics of the four deposits seem to indicate a large capping would be required for each deposit. To obtain a coefficient of variation of 1.2 or less, the capping value with respect to the ninety fifth (95%) percentile values range from 3.00 to 5.73 g while the 99% values range from 7.60 to 14.89 g/t. Neither of these two percentiles produces an acceptable capping value to Rusoro or Micon due to the style of mineralization.

The mineralization within the wireframes is not strictly high grade mineral lenses. Lower grade mineralization is included which most likely has affected the basic statistics and skewed conservatively. Also there is low grade and some high grade intervals outside the wireframes which correspond to footwall and hangingwall mineralized structure which can't be taken into account by the wireframe constraints.

It was then agreed to by Rusoro and Micon geologists that an overall cap value of 34.285 g/t Au (1 oz/t) would be adequate for the database. There were only 18 samples out of 59,245 that had to be capped.

14.2.2.7 Variography

The four composite semi-variograms were calculated for gold. The geological models of the deposit assisted in determining the major, semi-major and minor directions of continuity. The semi-variograms for each of the four known deposits in the Incredible 6 concession have been reproduced in Figure 14.8 and the parameters are summarized in Table 14.23.

Figure 14.8
Semi-Variograms for Increible 6 Deposits

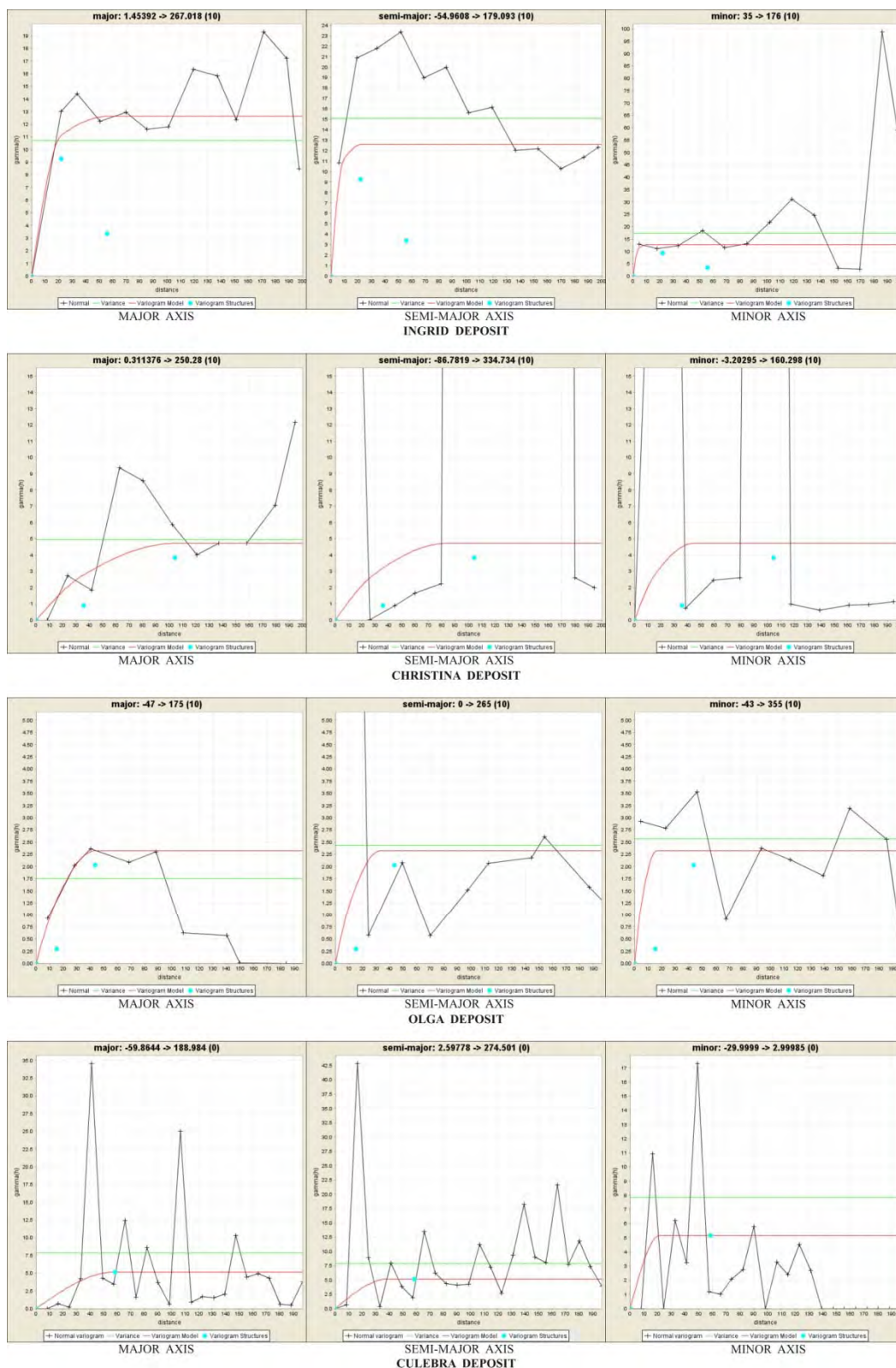


Table 14.23
Incredible 6 Variogram Parameters

Deposit		c0	c1	c2	c3	M:SM	M:m	Plunge	Dip
Ingrid	Variance	0.000	7.091	6.27	-				
	Range		30.90	55.18	-				
	M+I	28 m							
	Bearing	267.02°						1.45°	55.0°
	Ratio					2.357	5.513		
Christina	Variance	0.000	0.900	3.817	-				
	Range		35.62	104.40	-				
	M+I	58 m							
	Bearing	250.28°						0.31°	-86.8°
	Ratio					1.244	2.341		
Olga	Variance	0.000	0.293	2.029	-				
	Range		15.12	43.57	-				
	M+I	25 m							
	Bearing	175.0°						-47°	0.00°
	Ratio					1.315	2.695		
Culebra	Variance	0.000	0.431	2.215					
	Range		22.45	58.53					
	M+I	33 m							
	Bearing	188.98°						-89.86°	5.17°
	Ratio					1.496	2.712		

Note : M = Major Axis, SM = Semi-major axis and m = Minor axis

14.2.3 Block Models

A block model encompassing the Ingrid, Christina and Olga deposits (new3_ioc_bm.mdl) and the associated drill holes and composites was constructed. The block model geometry is documented in Table 14.24.

Table 14.24
Ingrid-Olga-Christina Block Model Geometry

	Y	X	Z
Minimum Coordinates	815900	620760	-500
Maximum Coordinates	818000	623060	+230
User Block Size	10	10	10
Minimum Block Size	2.5	2.5	2.5
	Bearing = 0.0°	Dip = 0.0°	Plunge = 0.0°

The total number of blocks was 5,239,346.

A separate block model encompassing the Culebra deposit (new3_cub_bm.mdl) and the associated drill holes and composite was constructed. This was required because of the two separate sets of topography, saprolite/saprock and saprock/fresh rock contact DTMs. The block model geometry is documented in Table 14.25.

Table 14.25
Culebra Block Model Geometry

	Y	X	Z
Minimum Coordinates	816950	623240	-170
Maximum Coordinates	817640	624520	+200
User Block Size	10	10	5
Minimum Block Size	2.5	2.5	1.25
	Bearing = 0.0°	Dip = 0.0°	Plunge = 0.0°

The total number of blocks was 1,775,997.

Three attributes were created for each block model and are listed in Table 14.26.

Table 14.26
Block Model Attributes

Attribute	Type	Decimal	Background	Description
class	character	NA	unestimated	measured, indicated, inferred
density	float	2	-1.00	rock density
gold	float	2	-1.00	grams per metric ton gold

14.2.4 Interpolation Method

The estimation method employed was an Inverse Distance Cubed (ID³) calculation using search ellipsoids based on the variography discussed in Section 17.2.2.7. This method was chosen because the Nearest Neighbour method tends to generate fewer tonnes at a higher grade while the Ordinary Kriging and Inverse Distance Squared methods tend to produce higher tonnes at a lower grade. The Inverse Distance Cubed method tends to produce numbers that fall between these other methods.

The estimates were constrained to individual geological wireframe models acting as hard boundaries. An individual block estimate would not accept composites outside of the wireframes or in adjacent lenses that make up the wireframe model.

The block model parameters are documented in Table 14.27.

14.2.5 Block Model Report

The user (maximum) block volume is 10 m (N-S) by 10 m (E-W) by 10 m (vertical) for the Ingrid-Olga-Christina deposits and 10 m (N-S) by 10 m (E-W) by 5 m (vertical) for the Culebra deposit. This equates to 1,000 and 500 m³, respectively. The specific gravity varies within the deposits due to amount of decomposition of the rock. The density is one of the attributes of the block model. The tonnage of a saprolite user block is 1,600 t while a saprock user block is 2,350 t and fresh rock is 2,700 t.

A block model report was generated for each deposit with respect to its resource category and grouped with respect to gold grade cut-offs.

Table 14.27
Incredible 6 Block Model Parameters

Deposit	Rotation (°Az)			Search Radius (m)			Composites		
	Bearing	Plunge	Dip	Major	Semi	Minor	Min	Max	DDH
Measured Resource									
Ingrid	267.0	+1.5	+55.0	9	3.8	1.6	3	10	2
Christina	250.0	+0.3	-86.8	10	8.0	4.3	3	10	2
Olga	175.0	-47.0	0.0	8	6.1	3.0	3	10	2
Culebra	189.0	-59.9	+5.2	8	5.3	2.9	3	10	2
Indicated Resource									
Ingrid	267.0	+1.5	+55.0	38	16.1	6.9	3	15	2
Christina	250.0	+0.3	-86.8	58	46.6	24.8	3	15	2
Olga	175.0	-47.0	0.0	35	26.6	13.0	3	15	2
Culebra	189.0	-59.9	+5.2	43	28.7	15.9	3	15	2
Inferred Resources									
Ingrid	267.0	+1.5	+55.0	300	127.3	54.4	3	15	6
Christina	250.0	+0.3	-86.8	300	241.2	128.2	3	15	6
Olga	175.0	-47.0	0.0	300	228.1	111.3	3	15	10
Culebra	189.0	-59.9	+5.2	300	200.5	110.6	3	15	10

Note : DDH is maximum per drill hole

14.2.6 Block Model Validation

The block model and accompanying drill hole database were compared visual in section (north-south and east-west), plan and in 3D. The block model has been constrained to the 3D wireframes. All blocks outside the wireframes have a grade attribute of -1. Visually the blocks and their respective grade attributes correspond well to both grade and 3D location of the mineralized intervals within the database.

The method used (Inverse Distance Cubed or ID³) normally produces tonnage and grade that is between the Nearest Neighbour and the Ordinary Kriging/Inverse Distance Squared (ID²) methods. A comparison of the previous 2006-7 estimate shows a slightly lower tonnage and lower grade. This difference in grade can be attributed to the different interpolation method, and search radii parameters used as well as more drilling and new wireframe constraints. Micon believes that the block model results portray the best estimate at this time of the mineralization in the four deposits that make up the Incredible 6 concession.

14.2.7 Incredible 6 Mineral Resource Estimate

14.2.7.1 Ingrid Deposit

Table 14.28 and Table 14.29 document the Ingrid deposit mineral resources with variable cut-off scenarios for the measured and indicated, and inferred resource categories, respectively.

Table 14.28
Ingrid Measured and Indicated Mineral Resource

Cut Off Grade	Measured			Indicated			Total M + I		
	Tonnes (millions)	Grade (g/t)	Au oz (000)	Tonnes (millions)	Grade (g/t)	Au oz (000)	Tonnes (millions)	Grade (g/t)	Au oz (000)
0.1	0.018	1.52	0.9	21.8	1.43	1,000.0	21.8	1.43	1,000.7
0.2	0.016	1.69	0.9	19.2	1.60	987.7	19.2	1.60	988.6
0.3	0.014	1.84	0.8	17.2	1.75	972.0	17.2	1.75	972.8
0.4	0.014	1.84	0.8	15.4	1.92	950.8	15.4	1.92	951.6
0.5	0.013	2.01	0.8	13.7	2.10	927.0	13.7	2.10	927.8
0.6	0.013	2.01	0.8	12.3	2.28	902.8	12.3	2.28	903.6
0.7	0.013	2.01	0.8	11.2	2.43	880.0	11.2	2.43	880.8
0.8	0.013	2.01	0.8	10.3	2.60	856.3	10.3	2.60	857.1
0.9	0.010	2.33	0.7	9.4	2.75	833.7	9.4	2.75	834.4
1.0	0.009	2.54	0.7	8.8	2.88	813.8	8.8	2.88	814.5

Table 14.29
Ingrid Inferred Mineral Resource

Cut-off	Inferred		
Grade	Tonnes	Grade	Au oz
(g/t)	(millions)	(g/t)	(000)
0.1	16.0	0.94	483.7
0.2	13.4	1.09	471.8
0.3	11.6	1.22	457.6
0.4	9.9	1.38	438.0
0.5	8.6	1.52	420.2
0.6	7.6	1.65	401.6
0.7	6.8	1.76	385.4
0.8	6.1	1.89	367.8
0.9	5.4	2.01	350.7
1.0	4.9	2.12	334.6

1. It cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.
2. Mineral resources which are not mineral reserves do not have demonstrated economic viability.

At the selected cut-off grade of 0.5 g/t Au, the Ingrid deposit contains a measured plus indicated mineral resource of 13.7 million tonnes grading 2.10 g/t Au, and an inferred mineral resource of 8.6 million tonnes grading 1.52 g/t Au.

14.2.7.2 Christina Deposit

Table 14.30 and Table 14.31 document the Christina deposit mineral resources with variable cut-off scenarios for the measured and indicated, and inferred resource categories, respectively.

Table 14.30
Christina Measured and Inferred Mineral Resource

Cut Off Grade	Measured			Indicated			Total M + I		
	Tonnes (millions)	Grade (g/t)	Au oz (000)	Tonnes (millions)	Grade (g/t)	Au oz (000)	Tonnes (millions)	Grade (g/t)	Au oz (000)
0.1	0.459	1.22	18.0	4.605	0.74	109.6	5.064	0.78	127.6
0.2	0.381	1.44	17.6	3.351	0.96	103.7	3.732	1.01	121.3
0.3	0.323	1.65	17.1	2.656	1.15	98.2	2.979	1.20	115.4
0.4	0.282	1.84	16.7	2.249	1.30	93.7	2.531	1.36	110.4
0.5	0.250	2.02	16.2	1.871	1.47	88.3	2.120	1.53	104.5
0.6	0.222	2.20	15.7	1.586	1.63	83.3	1.808	1.70	99.0
0.7	0.195	2.42	15.2	1.405	1.76	79.5	1.600	1.84	94.6
0.8	0.170	2.66	14.6	1.231	1.90	75.3	1.401	1.99	89.8
0.9	0.160	2.78	14.3	1.083	2.05	71.2	1.243	2.14	85.5
1.0	0.148	2.92	13.9	0.959	2.19	67.4	1.107	2.29	81.3

Table 14.31
Christina Inferred Mineral Resource

Cut-off	Inferred		
Grade	Tonnes	Grade	Au oz
(g/t)	(millions)	(g/t)	(000)
0.1	0.122	0.28	1.1
0.2	0.040	0.60	0.8
0.3	0.023	0.87	0.6
0.4	0.021	0.90	0.6
0.5	0.020	0.92	0.6
0.6	0.019	0.94	0.6
0.7	0.013	1.09	0.4
0.8	0.010	1.19	0.4
0.9	0.007	1.33	0.3
1.0	0.005	1.49	0.2

- 1) It cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.
- 2) Mineral resources which are not mineral reserves do not have demonstrated economic viability.

At the selected cut-off grade of 0.5 g/t Au, the Christine deposit contains a measured plus indicated mineral resource of 2.12 million tonnes grading 1.53 g/t Au, and an inferred mineral resource 0.02 million tonnes grading 0.92 g/t Au, at a 0.5 g/t Au cut-off.

14.2.7.3 Olga Deposit

The following two tables document the Olga deposit mineral resources with variable cut-off scenarios for the measured and indicated, and inferred resource categories, respectively.

Table 14.32
Olga Measured and Inferred Mineral Resource

Cut Off Grade	Measured			Indicated			Total M + I		
	Tonnes (millions)	Grade (g/t)	Au oz (000)	Tonnes (millions)	Grade (g/t)	Au oz (000)	Tonnes (millions)	Grade (g/t)	Au oz (000)
0.1	0.037	1.31	1.6	1.600	0.60	31.0	1.637	0.62	32.5
0.2	0.030	1.62	1.5	1.219	0.74	29.2	1.249	0.77	30.7
0.3	0.028	1.70	1.5	0.985	0.86	27.3	1.013	0.89	28.9
0.4	0.020	2.19	1.4	0.762	1.01	24.8	0.782	1.04	26.2
0.5	0.017	2.58	1.4	0.571	1.21	22.1	0.587	1.25	23.5
0.6	0.014	2.93	1.3	0.480	1.33	20.5	0.495	1.38	21.9
0.7	0.013	3.18	1.3	0.416	1.44	19.2	0.429	1.49	20.5
0.8	0.013	3.18	1.3	0.346	1.57	17.5	0.359	1.63	18.8
0.9	0.011	3.55	1.3	0.283	1.74	15.8	0.294	1.80	17.1
1.0	0.010	3.79	1.2	0.241	1.87	14.5	0.251	1.95	15.8

Table 14.33
Olga Inferred Mineral Resource

Cut-off	Inferred		
Grade (g/t)	Tonnes (millions)	Grade (g/t)	Au oz (000)
0.1	1.486	0.33	15.9
0.2	0.904	0.46	13.2
0.3	0.571	0.58	10.6
0.4	0.313	0.77	7.7
0.5	0.230	0.88	6.5
0.6	0.175	0.99	5.6
0.7	0.139	1.07	4.8
0.8	0.112	1.15	4.1
0.9	0.091	1.22	3.6
1.0	0.069	1.31	2.9

1. It cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.
2. Mineral resources which are not mineral reserves do not have demonstrated economic viability.

At the selected cut-off grade of 0.5 g/t Au, the Olga deposit contains a measured plus indicated mineral resource of 0.587 million tonnes grading 1.25 g/t Au, and an inferred mineral resource of 0.230 million tonnes grading 0.88 g/t Au.

14.2.7.4 Culebra Deposit

Table 14.34 and Table 14.35 document the Christina deposit mineral resources with variable cut-off scenarios for the measured and indicated, and inferred resource categories, respectively.

Table 14.34
Culebra Measured and Indicated Mineral Resource

Cut Off Grade	Measured			Indicated			Total M + I		
	Tonnes (millions)	Grade (g/t)	Au oz (000)	Tonnes (millions)	Grade (g/t)	Au oz (000)	Tonnes (millions)	Grade (g/t)	Au oz (000)
0.1	0.264	1.69	14.4	10.050	1.05	339.8	10.314	1.07	354.2
0.2	0.235	1.89	14.3	8.507	1.22	332.3	8.742	1.23	346.6
0.3	0.214	2.05	14.1	7.517	1.34	324.4	7.730	1.36	338.4
0.4	0.195	2.21	13.9	6.593	1.48	314.0	6.788	1.50	327.8
0.5	0.181	2.35	13.7	5.826	1.62	302.9	6.008	1.64	316.6
0.6	0.168	2.49	13.4	5.241	1.74	292.5	5.409	1.76	306.0
0.7	0.158	2.61	13.2	4.628	1.88	279.7	4.786	1.90	293.0
0.8	0.152	2.67	13.1	4.138	2.01	267.9	4.290	2.04	281.0
0.9	0.133	2.94	12.6	3.686	2.16	255.5	3.819	2.18	268.1
1.0	0.127	3.03	12.4	3.318	2.29	244.3	3.446	2.32	256.7

Table 14.35
Culebra Inferred Mineral Resource

Cut-off	Inferred		
Grade	Tonnes	Grade	Au oz
(g/t)	(millions)	(g/t)	(000)
0.1	1.622	0.75	39.1
0.2	1.377	0.86	37.9
0.3	1.154	0.97	36.1
0.4	0.915	1.14	33.4
0.5	0.706	1.34	30.4
0.6	0.562	1.54	27.9
0.7	0.452	1.76	25.6
0.8	0.377	1.96	23.8
0.9	0.326	2.14	22.4
1.0	0.275	2.36	20.8

1. It cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.
2. Mineral resources which are not mineral reserves do not have demonstrated economic viability.

At the selected cut-off grade of 0.5 g/t Au, the Culebra deposit contains a measured plus indicated mineral resource of 6.008 million tonnes grading 1.64 g/t Au, and an inferred mineral resource of 0.706 million tonnes grading 1.34 g/t Au.

14.2.7.5 Incredible 6 Mineral Resources

The mineral resources of the four major deposits within the Incredible 6 concession are listed below at the 0.5 g/t Au cut-off. These mineral resources are inclusive of any mineral reserves on the property. Mineral resources that are not mineral reserves do not have demonstrated economic viability. As of the effective date of this report, there are no known legal, political, environmental or other risks that could materially affect the potential development of the mineral resources and reserves.

Table 14.36
Incredible 6 Mineral Resource at 0.5 g/t Au Cut-off

Deposit	Measured			Indicated			Total M + I			Inferred		
	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Elisa/Ingrid	0.01	2.01	0.8	13.70	2.10	927.0	13.72	2.10	927.8	8.62	1.52	420.2
Christina	0.25	2.02	16.2	1.87	1.47	88.3	2.12	1.53	104.5	0.02	0.92	0.6
Olga/Enoc	0.02	2.58	1.4	0.57	1.21	22.1	0.59	1.25	23.5	0.23	0.88	6.5
Culebra	0.18	2.35	13.7	5.83	1.62	302.9	6.01	1.64	316.6	0.71	1.34	30.3
TOTAL	0.46	2.17	32.1	21.97	1.90	1,340.3	22.43	1.90	1,372.4	9.58	1.49	457.7

1. It should be noted that the numbers in the table are rounded off and may not add up.
2. It cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.
3. Mineral resources which are not mineral reserves do not have demonstrated economic viability.

There has been a decrease in the overall tonnes and grade of the four deposits although in most cases this decrease is relatively small. Reasons for this decrease are as follows:

- Difference in interpolation method (i.e. ID² versus ID³);
- Search radii differences (i.e. anisotropy ratio, ellipsoid orientation, search axis distances, etc.);
- Maximum/minimum number of samples used in the interpolation;
- Maximum number of samples per hole;
- Wireframe interpretation and construction;
- Additional drill hole data and composites (2007, 2008).

The differences are documented in Table 14.37.

Table 14.37
Incredible 6 Mineral Resource Comparison

Deposit	Cut-Off Grade (g/t Au)	M + I		Inferred	
		Tonnes (millions)	Au (oz) (000)	Tonnes (millions)	Au (oz) (000)
Ingrid	0.5	- 1.362	- 128.2	- 3.215	- 384.3
Christina	0.5	- 0.460	- 70.2	- 1.910	- 99.2
Olga	0.5	+ 0.347	+ 8.8	- 1.090	- 67.6
Culebra	0.5	+ 0.458	- 26.6	-1.744	- 93.2
TOTAL	0.5	- 1.016	- 216.2	- 7.958	- 644.3

15.0 MINERAL RESERVE ESTIMATES

As of December 31, 2010, the mineral reserves for the Choco 10 and Incredible 6 deposits are as summarized in Table 15.1 and Table 15.2 respectively. Mining losses of 1% were assumed, and dilution factors of 11% and 14% were applied to oxide ore and transition/fresh ore respectively. Ore tonnes were diluted by an average grade of 0.10 g/t. These mineral reserves are included within the mineral resources stated in Section 14.

Table 15.1
Choco 10 Mineral Reserves (December 31, 2010)

Deposit	COG	Proven			Probable			Total P+P		
		(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Rosika	0.4	1.95	1.88	118	8.15	2.10	549	10.10	2.05	667
Coacia	0.4	0.18	1.28	8	0.61	1.39	27	0.79	1.36	35
Pisolita	0.4	0.07	1.35	3	1.26	1.91	78	1.33	1.88	81
VBK	0.4	28.66	1.82	1,676	45.77	1.36	2,005	74.43	1.54	3,681
TOTAL	0.4	30.86	1.82	1,804	55.79	1.48	2,659	86.65	1.60	4,463

Table 15.2
Incredible 6 Mineral Reserves (December 31, 2010)

Deposit	COG	Proven			Probable			Total P+P		
		(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)	(Mt)	Grade (g/t)	Au oz (000)
Culebra	0.4	0.17	2.23	12	3.49	1.69	190	3.65	1.72	202
Christina	0.4	0.09	2.40	7	0.67	1.65	36	0.76	1.74	43
Elisa/Ingrid	0.4	0.01	2.02	1	4.81	2.03	314	4.82	2.03	315
Olga/Enoc	0.4	0.00	0.00	0	0.00	0	0	0.00	0.00	0
TOTAL	0.4	0.27	2.28	20	8.97	1.87	540	9.24	1.88	559

The mine design work supporting these reserves is described more fully in Section 16.

16.0 MINING METHODS

16.1 MINING OPERATIONS

The Rosika, Coacia and Pisolita deposits have been mined for at least 6 years. The current mill has a capacity of 5,000 t/d. It is proposed to increase the plant capacity to 20,000 t/d. This increase in capacity is based on continuing to mine the combined Rosika/Coatia/Pisolita pits, bringing the large VBK open pit on line as well as developing the Incredible 6 deposits.

16.2 MINE DESIGN

The feasibility study considers mining from the Choco 10 (Rosika, Coacia, Pisolita and VBK) and the Incredible 6 (Culebra, Christina, Elisa) deposits at a combined rate of 20,000 t/d or 7.3 Mt/y of mill feed. New resource block models generated by Micon were used as the basis for the open pit design. The mine design process involved the following steps:

- Generation of economically-optimized ultimate open pit shells for each deposit using Whittle 4X software
- Design of phased pits with access ramps for each deposit. The tonnage within this minable pit serves as the basis for the mineral reserve.
- Selection of the fleet of drilling, loading, hauling and auxiliary equipment required to meet the design schedule.
- Estimation of mine capital expenditures and operating costs to the level of accuracy appropriate for a feasibility study.

16.2.1 Open Pit Optimization

Open pit optimization was completed using Gemcom Software's Whittle 4X open pit optimization program. This program uses the Lerchs-Grossman algorithm to determine the optimal economic open pit footprint for a given mineral resource. Once this footprint has been established, the software allows the development of a simple production schedule. For the Choco Feasibility Study, instead of the typical single block model, three different block models needed to be optimized separately to provide design limits.

The Choco Feasibility Study considers three gold deposits as the basis for the resource. These include Choco 10, Incredible 6, and Culebra. Choco 10 is made up of two different deposits, RCP and VBK. The other deposits are defined in individual block models for a total of three block models. All of the models were delivered to Micon in an ASCII format. These block models were imported into a Vulcan format and then exported into Whittle. Three different rock types (oxide, transition, and fresh) and average gold value above a cut-off were exported into the Whittle model. Pit slopes used in the Whittle analysis take into account the preliminary pit slope recommendations given by Knight Piésold in February, 2010. The recommended inter-ramp angles are 40° in saprolite/saprock and transition, and range between 47° and 51° in fresh rock.

For a mineral reserve estimate to comply with CIM definitions, no value should be attributed to inferred mineral resources in the design of an open pit. For this Feasibility Study, only measured and indicated materials could be considered as potential mineral resource in the Whittle pit optimizations. Inferred mineral resource was considered as waste only. The criteria used in the pit optimization are shown in Table 16.1, Table 16.2 and Table 16.3.

Table 16.1
Parameters used for the Choco 10 Whittle Pit Optimization

Area	Units	\$/Unit	Source
Ore Mining	US\$/Ore Tonne	\$3.000	Micon Scoping Study
Waste Mining	US\$/Waste Tonne	\$3.000	Micon Scoping Study
Processing	US\$/Ore Tonne	\$4.340	Micon Scoping Study
G&A	US\$/Ore Tonne	\$2.670	Micon Scoping Study
Contingency	US\$/Ore Tonne	\$2.370	Micon Scoping Study
Gold Price	US\$/Ounce	\$900.00	C. Jacobs 7/2010
Royalty	% of Gross Sales	3.0%	Micon Scoping Study
Selling Cost	US\$/Recovered Ounce Gold	\$94.50	Micon Scoping Study
Gold Recovery	% of Feed Grade	90.0%	Micon Scoping Study

Table 16.2
Parameters used for the Incredible 6 Whittle Pit Optimization

Area	Units	\$/Unit	Source
Ore Mining	US\$/Ore Tonne	\$3.000	Micon Scoping Study
Waste Mining	US\$/Waste Tonne	\$3.000	Micon Scoping Study
Processing	US\$/Ore Tonne	\$4.340	Micon Scoping Study
G&A	US\$/Ore Tonne	\$2.670	Micon Scoping Study
Contingency	US\$/Ore Tonne	\$2.370	Micon Scoping Study
Gold Price	US\$/Ounce	\$900.00	C. Jacobs 7/2010
Royalty	% of Gross Sales	3.0%	Micon Scoping Study
Selling Cost	US\$/Recovered Ounce Gold	\$94.50	Micon Scoping Study
Gold Recovery	% of Feed Grade	90.0%	Micon Scoping Study

Table 16.3
Parameters used for the Culebra Whittle Pit Optimization

Area	Units	\$/Unit	Source
Ore Mining	US\$/Ore Tonne	\$3.000	Micon Scoping Study
Waste Mining	US\$/Waste Tonne	\$3.000	Micon Scoping Study
Processing	US\$/Ore Tonne	\$4.340	Micon Scoping Study
G&A	US\$/Ore Tonne	\$2.670	Micon Scoping Study
Contingency	US\$/Ore Tonne	\$2.370	Micon Scoping Study
Gold Price	US\$/Ounce	\$900.00	C. Jacobs 7/2010
Royalty	% of Gross Sales	3.0%	Micon Scoping Study
Selling Cost	US\$/Recovered Ounce Gold	\$94.50	Micon Scoping Study
Gold Recovery	% of Feed Grade	90.0%	Micon Scoping Study

The topographic base for each block model/Whittle model was provided by Rusoro and is based on current topography in November, 2010.

For all block models, mining dilution and mining recovery was input into the Whittle open pit optimization. Dilution in Whittle is assumed to be 11% in oxide rock and 14% in

transition and fresh rock. Mining recovery in Whittle is assumed to be 99% in all material types. These dilution and mining recoveries were provided by Rusoro and is based on actual operating experience at Choco 10.

Each of the block models were loaded into Whittle and reblocked to a standard block size (40 x 40 x 10 m). Whittle optimization was performed on each block model (in the order of Choco 10, Increible 6, and Culebra). Using these results a pit shell at the full gold price was selected and used as a maximum pit limit for Whittle Milawa scheduling.

Scheduling using the Whittle Milawa algorithm to achieve the required processing plant feed and maximum waste movement for each period was then carried out. Milawa production scheduling is the method, which is used by Whittle to take the results of a pit optimization and generate a balanced production schedule.

In order to accomplish this, a series of pit shells are selected at various increments representing lower to higher gold prices. The program then attempts to balance the production schedule on total material movement while filling the processing plant and producing the maximum allowable gold ounces. This generates a balanced and more realistic mining production schedule.

16.2.1.1 Whittle Open Pit Optimization Results

The Whittle open pit optimization work completed for the Feasibility Study considered an annual production level of 7.3 million tonnes. These schedules were run at various gold prices in order to determine the sensitivity of the various deposits to changing economic conditions. The results for each block model are shown below in Table 16.4, Table 16.5 and Table 16.6.

Based on this work, for the Choco 10 deposit pit shell 49 was selected for design, for the Increible 6 deposit pit shell 44 was selected, and for the Culebra deposit pit shell 41 was selected. These three pit shells were then used as the basis for the more detailed feasibility design.

Table 16.4
Whittle Results – Choco 10

Scenario – Gold Price (\$/oz)	Class	Gold Price (US\$/oz)	Mill			Total Waste Tonnes	Total Tonnes	Stripping Ratio	Whittle Estimated NPV (No CAPEX, US\$)
			Resource Tonnes	Au (g/t)	Rec'd ozs Au				
\$500	M+I	\$500	11,061,000	3.030	970,000	42,009,000	53,070,000	3.80	\$98,615,000
\$550	M+I	\$550	12,688,000	2.869	1,053,000	48,250,000	60,938,000	3.80	\$138,261,000
\$600	M+I	\$600	54,423,000	2.255	3,551,000	250,660,000	305,083,000	4.61	\$129,426,000
\$650	M+I	\$650	61,353,000	2.150	3,817,000	267,456,000	328,809,000	4.36	\$218,618,000
\$700	M+I	\$700	69,798,000	2.035	4,110,000	287,275,000	357,073,000	4.12	\$303,937,000
\$750	M+I	\$750	79,480,000	1.938	4,458,000	318,384,000	397,864,000	4.01	\$374,703,000
\$800	M+I	\$800	84,923,000	1.874	4,606,000	327,760,000	412,683,000	3.86	\$468,451,000
\$850	M+I	\$850	99,858,000	1.796	5,190,000	401,186,000	501,044,000	4.02	\$505,400,000
\$900	M+I	\$900	104,622,000	1.743	5,277,000	403,939,000	508,561,000	3.86	\$594,362,000
\$950	M+I	\$950	118,857,000	1.659	5,706,000	460,231,000	579,088,000	3.87	\$628,312,000
\$1000	M+I	\$1,000	138,691,000	1.580	6,341,000	561,993,000	700,684,000	4.05	\$601,459,000
\$1050	M+I	\$1,050	146,289,000	1.528	6,469,000	568,127,000	714,416,000	3.88	\$621,009,000
\$1100	M+I	\$1,100	153,850,000	1.490	6,634,000	589,165,000	743,015,000	3.83	\$749,650,000
\$1150	M+I	\$1,150	159,047,000	1.466	6,747,000	606,090,000	765,137,000	3.81	\$829,538,000
\$1200	M+I	\$1,200	166,667,000	1.433	6,912,000	630,998,000	797,665,000	3.79	\$827,840,000
\$1250	M+I	\$1,250	171,704,000	1.410	7,006,000	644,886,000	816,590,000	3.76	\$917,198,000
\$1300	M+I	\$1,300	178,068,000	1.373	7,075,000	644,543,000	822,611,000	3.62	\$1,006,407,000
\$1350	M+I	\$1,350	181,350,000	1.355	7,111,000	645,754,000	827,104,000	3.56	\$1,081,296,000
\$1400	M+I	\$1,400	185,557,000	1.332	7,153,000	646,024,000	831,581,000	3.48	\$1,157,599,000
\$1450	M+I	\$1,450	189,666,000	1.314	7,212,000	653,773,000	843,439,000	3.45	\$1,229,961,000
\$1500	M+I	\$1,500	193,138,000	1.296	7,244,000	653,527,000	846,665,000	3.38	\$1,309,273,000

Table 16.5
Whittle Results – Incredible 6

Scenario – Gold Price (\$/oz)	Class	Gold Price (US\$/oz)	Mill			Total Waste Tonnes	Total Tonnes	Stripping Ratio	Whittle Estimated NPV (No CAPEX, US\$)
			Resource Tonnes	Au (g/t)	Rec'd ozs Au				
\$500	M+I	\$500	1,392,000	3.254	131,000	6,095,000	7,487,000	4.38	\$15,458,000
\$550	M+I	\$550	1,926,000	3.116	174,000	9,642,000	11,568,000	5.01	\$23,007,000
\$600	M+I	\$600	2,557,000	2.888	214,000	13,099,000	15,656,000	5.12	\$32,309,000
\$650	M+I	\$650	2,912,000	2.724	230,000	14,348,000	17,260,000	4.93	\$42,497,000
\$700	M+I	\$700	3,036,000	2.672	235,000	14,786,000	17,822,000	4.87	\$53,301,000
\$750	M+I	\$750	3,536,000	2.519	258,000	17,372,000	20,908,000	4.91	\$64,488,000
\$800	M+I	\$800	3,899,000	2.424	274,000	19,312,000	23,211,000	4.95	\$76,448,000
\$850	M+I	\$850	4,236,000	2.351	288,000	21,381,000	25,617,000	5.05	\$89,037,000
\$900	M+I	\$900	4,734,000	2.254	309,000	24,498,000	29,232,000	5.17	\$102,076,000
\$950	M+I	\$950	6,072,000	2.105	370,000	35,381,000	41,453,000	5.83	\$115,595,000
\$1000	M+I	\$1,000	6,568,000	2.063	392,000	39,572,000	46,140,000	6.02	\$132,064,000
\$1050	M+I	\$1,050	7,127,000	2.026	418,000	45,021,000	52,148,000	6.32	\$148,917,000
\$1100	M+I	\$1,100	7,261,000	2.005	421,000	45,591,000	52,852,000	6.28	\$167,164,000
\$1150	M+I	\$1,150	7,442,000	1.973	425,000	46,049,000	53,491,000	6.19	\$185,706,000
\$1200	M+I	\$1,200	7,539,000	1.956	427,000	46,299,000	53,838,000	6.14	\$204,457,000
\$1250	M+I	\$1,250	7,693,000	1.927	429,000	46,483,000	54,176,000	6.04	\$223,281,000
\$1300	M+I	\$1,300	8,191,000	1.873	444,000	50,140,000	58,331,000	6.12	\$241,999,000
\$1350	M+I	\$1,350	8,545,000	1.837	454,000	52,897,000	61,442,000	6.19	\$261,182,000
\$1400	M+I	\$1,400	9,124,000	1.810	478,000	60,261,000	69,385,000	6.60	\$280,732,000
\$1450	M+I	\$1,450	9,468,000	1.772	486,000	62,152,000	71,620,000	6.56	\$301,254,000
\$1500	M+I	\$1,500	9,965,000	1.731	499,000	66,196,000	76,161,000	6.64	\$321,707,000

Table 16.6
Whittle Results – Culebra

Scenario – Gold Price (\$/oz)	Class	Gold Price (US\$/oz)	Mill			Total Waste Tonnes	Total Tonnes	Stripping Ratio	Whittle Estimated NPV (No CAPEX, US\$)
			Resource Tonnes	Au (g/t)	Rec'd ozs Au				
\$500	M+I	\$500	924,000	3.164	85,000	3,215,000	4,139,000	3.48	\$11,813,000
\$550	M+I	\$550	1,344,000	2.752	107,000	4,621,000	5,965,000	3.44	\$16,211,000
\$600	M+I	\$600	1,469,000	2.643	112,000	4,925,000	6,394,000	3.35	\$21,409,000
\$650	M+I	\$650	2,010,000	2.378	138,000	7,148,000	9,158,000	3.56	\$27,107,000
\$700	M+I	\$700	2,581,000	2.107	157,000	8,415,000	10,996,000	3.26	\$33,688,000
\$750	M+I	\$750	2,779,000	2.020	162,000	8,630,000	11,409,000	3.11	\$41,086,000
\$800	M+I	\$800	2,930,000	1.961	166,000	8,824,000	11,754,000	3.01	\$48,707,000
\$850	M+I	\$850	3,064,000	1.925	171,000	9,303,000	12,367,000	3.04	\$56,480,000
\$900	M+I	\$900	3,298,000	1.854	177,000	9,923,000	13,221,000	3.01	\$64,344,000
\$950	M+I	\$950	3,460,000	1.815	182,000	10,554,000	14,014,000	3.05	\$72,542,000
\$1000	M+I	\$1,000	3,680,000	1.757	187,000	11,132,000	14,812,000	3.03	\$80,917,000
\$1050	M+I	\$1,050	3,917,000	1.696	192,000	11,728,000	15,645,000	2.99	\$89,383,000
\$1100	M+I	\$1,100	4,033,000	1.659	194,000	11,685,000	15,718,000	2.90	\$98,128,000
\$1150	M+I	\$1,150	4,102,000	1.642	195,000	11,796,000	15,898,000	2.88	\$106,982,000
\$1200	M+I	\$1,200	4,298,000	1.599	199,000	12,399,000	16,697,000	2.88	\$115,769,000
\$1250	M+I	\$1,250	4,540,000	1.548	203,000	13,034,000	17,574,000	2.87	\$124,624,000
\$1300	M+I	\$1,300	4,932,000	1.506	215,000	15,740,000	20,672,000	3.19	\$133,730,000
\$1350	M+I	\$1,350	5,066,000	1.479	217,000	15,925,000	20,991,000	3.14	\$143,302,000
\$1400	M+I	\$1,400	5,372,000	1.440	224,000	17,624,000	22,996,000	3.28	\$152,574,000
\$1450	M+I	\$1,450	5,638,000	1.411	230,000	19,280,000	24,918,000	3.42	\$162,261,000
\$1500	M+I	\$1,500	5,756,000	1.400	233,000	20,095,000	25,851,000	3.49	\$172,490,000

16.3 LIFE OF MINE PIT DESIGN

The open pit designs are consistent with inter-ramp and overall slope angles recommended by Knight Piésold in February, 2010. In saprolite/saprock 10 m bench height, 6 m berm and 60° face angle gives an inter-ramp slope of 40°. In fresh rock, 20 m bench height, 8 m berm and 60° face angle gives an inter-ramp slope <46°, versus a recommended inter-ramp slope range of 47-51°. As recommended, the overall slope in VBK pit does not exceed 43°. Haulage roads were typically designed to a width of 25 m to accommodate two-way haulage. However, as shown in Figure 16.1 and Figure 16.2, one-way haulage was designed for each of the smaller and shallower satellite pits.

16.4 MINE PRODUCTION SCHEDULE

The mine production schedule, summarized in Table 16.7, was developed using phased pit designs and Minesched 7.0 scheduling software. Numerous constraints were set to meet the following objectives:

- Balance the stripping profile to maximize utilization of the truck fleet (minimizing the need to purchase new trucks later in the project life);
- Manage ore stockpiles to ensure adequate ore feed is available (equivalent to 3 or 4 months of production);
- Ensure that 3 to 4 mining faces are available at all times for operational flexibility;
- Maintain higher ore grades prior to and during mill ramp up.

Table 16.7
Mine Production Schedule

Period	Proven+ Probable Ore (000 t)	Waste (000 t)	Total (000 t)	Milled ¹ (000 t)	Grade Au (g/t)	Ore Stockpile (000 t)
2011	2,179	13,018	15,196	1,825	1.59	477
2012	2,635	21,959	24,593	1,825	1.52	1,383
2013	2,393	31,192	33,584	1,825	1.75	2,019
2014	6,268	55,556	61,824	6,342	1.86	1,922
2015	7,002	60,490	67,493	7,300	1.82	1,573
2016	6,692	48,955	55,647	7,300	1.20	873
2017	7,423	22,142	29,565	7,300	1.26	997
2018	7,232	21,019	28,251	7,300	1.37	907
2019	7,499	17,467	24,966	7,300	1.60	1,115
2020	7,966	20,760	28,726	7,300	1.58	1,781
2021	7,486	8,939	16,425	7,300	1.59	1,975
2022	7,206	9,219	16,425	7,300	1.66	1,855
2023	6,524	9,901	16,425	7,300	2.25	1,004
2024	6,601	9,869	16,470	7,300	1.76	305
2025	7,004	8,107	15,111	7,300	1.61	9
2026	3,776	2,441	6,217	3,844	1.66	0
TOTAL	95,886	361,034	456,918	95,961	1.63	-

1. Milled tonnage includes a stockpile balance of 75,000 tonnes brought forward from 2010.

Figure 16.1
Life of Mine Pit Design – Choco 10 (VBK-Rosika-Coacia-Pisolita)

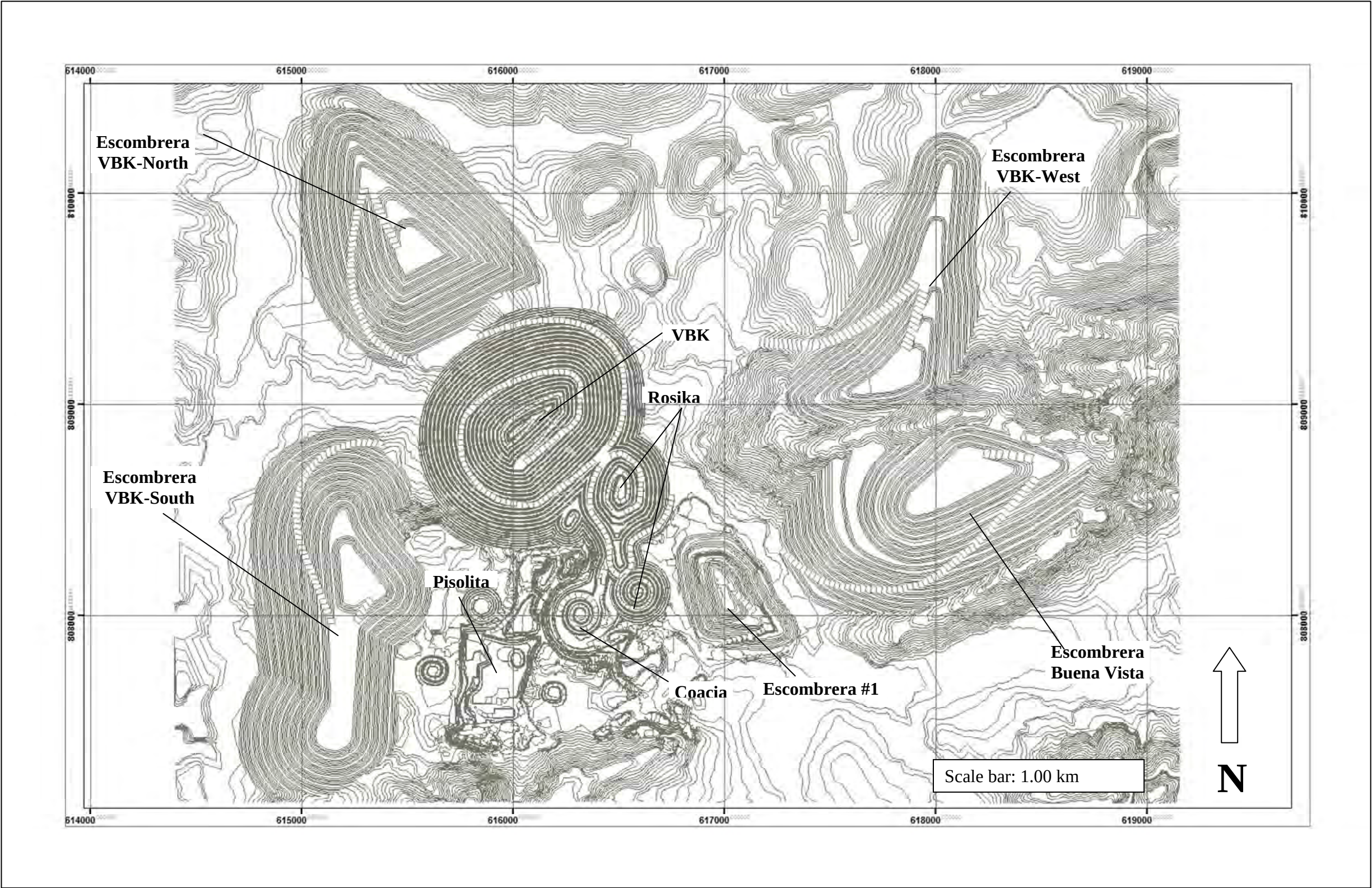
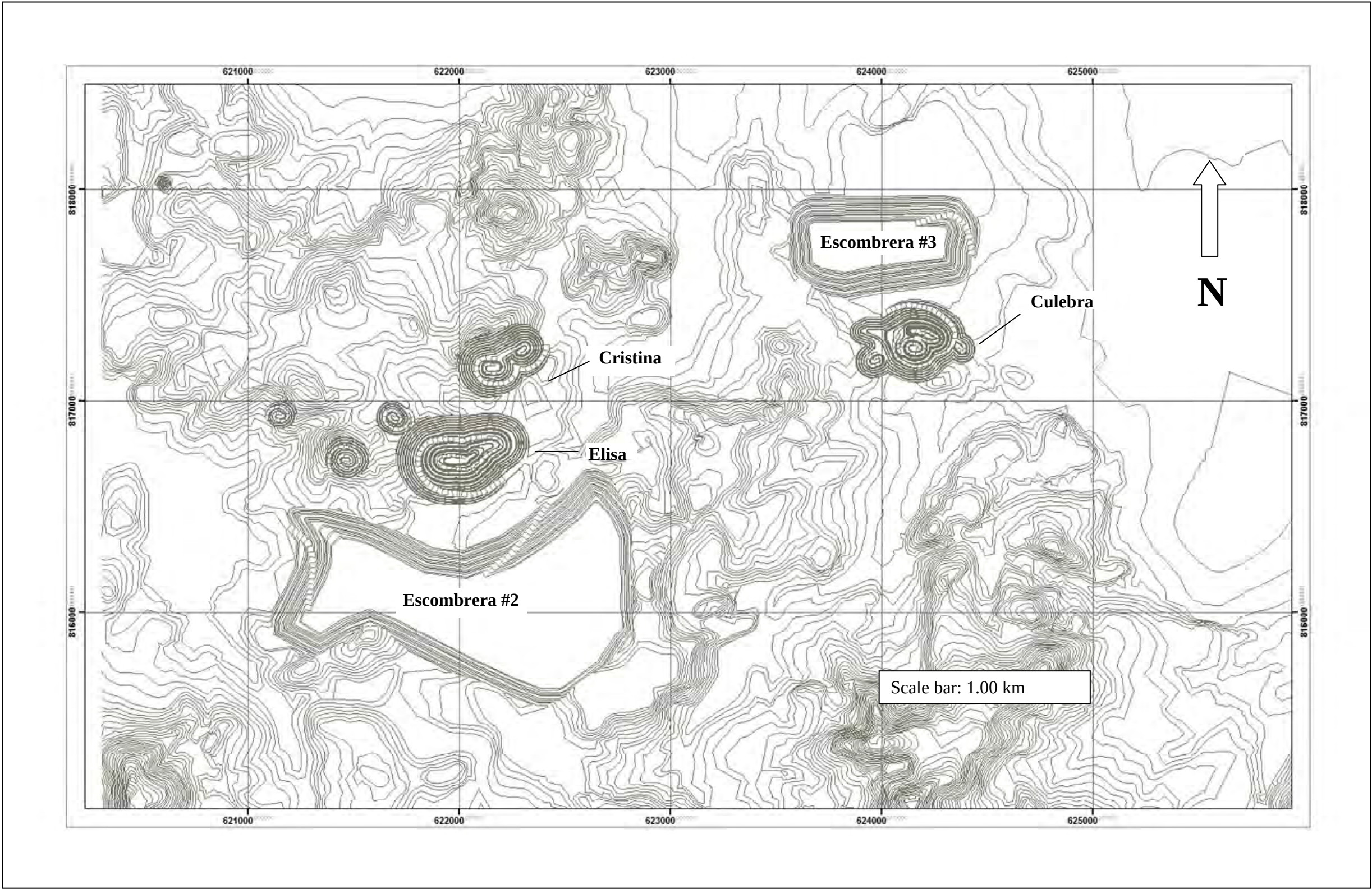


Figure 16.2
Life of Mine Pit Design – Incredible 6 (Culebra-Cristina-Elisa)



16.5 MINING EQUIPMENT

The existing fleet, operated by a local mining contractor, was assumed to remain in operation to feed the mill at a rate of approximately 5000t/d through 2012 Q2. Beginning in 2012 Q2, larger equipment would be phased in and a mixed fleet would remain in operation through 2012 until the new fleet is completely mobilized by 2013 Q1.

The required number of equipment units was determined from the annual production schedule for ore and waste with allowances for availability and utilization based on experience. Drill requirements were based on assessed penetration rates and shovel requirements were based on average cycle times and hourly productivities. Truck fleet requirements were estimated using annual haulage profiles. Auxiliary equipment requirements were based on experience.

The annual fleet requirements for the Choco 10 and Incredible 6 complex are shown in Table 16.8.

Table 16.8
Annual Mine Equipment Requirements

Year	CAT 793F Truck	Bucyrus RH-200 Shovel	CAT 994F Loader	AtlasCopco ROC-L8 Drill	CAT D10T Dozer	CAT 854K Dozer	CAT 16M Grader
2011	0	0	0	0	0	0	0
2012	10	2	1	3	1	1	1
2013	13	2	1	4	3	2	2
2014	23	3	2	8	3	2	2
2015	25	3	2	8	3	2	2
2016	25	3	2	7	3	2	2
2017	15	2	1	3	3	2	2
2018	15	2	1	3	3	2	2
2019	14	2	1	3	3	2	2
2020	13	2	1	2	3	2	2
2021	10	1	1	2	3	2	2
2022	10	1	1	2	3	2	2
2023	10	1	1	2	3	2	2
2024	10	1	1	2	3	2	2
Max Units Req'd	25	3	2	8	3	2	2

As shown in Table 16.9, no replacement purchases of trucks, face shovels, production loaders and drills will be necessary as the material movement profiles were intentionally balanced to maximize utilization of equipment hours and minimize the need for replacement purchases late in the project life. However, replacement purchases for tracked dozers, wheeled dozers and road graders are anticipated in 2020.

Table 16.9
Equipment Replacement Requirements

Year	CAT 793F Truck	Bucyrus RH-200 Shovel	CAT 994F Loader	AtlasCopco ROC-L8 Drill	CAT D10T Dozer	CAT 854K Dozer	CAT 16M Grader
2011	0	0	0	0	0	0	0
2012	0	0	0	0	0	0	0
2013	0	0	0	0	0	0	0
2014	0	0	0	0	0	0	0
2015	0	0	0	0	0	0	0
2016	0	0	0	0	0	0	0
2017	0	0	0	0	0	0	0
2018	0	0	0	0	0	0	0
2019	0	0	0	0	0	0	0
2020	0	0	0	2	3	2	2
2021	0	0	0	0	0	0	0
2022	0	0	0	0	0	0	0
2023	0	0	0	0	0	0	0
2024	0	0	0	0	0	0	0
Max Units Req'd	0	0	0	2	3	2	2

16.6 WASTE ROCK STORAGE

Waste storage dumps designs provided by Rusoro were used as starting basis for the RCP, VBK and Incredible 6 deposits. For Incredible 6, no changes to the waste rock designs were necessary as adequate capacity was available with the existing designs. However, additional capacity was required for the VBK and RCP deposits. Additional lifts were added to the original VBK and RCP dump designs, and two new dumps were designed as shown in Figure 16.1.

Waste storage for the RCP pits are located in two dumps to the east of the existing and planned ultimate pits. (Note: Escombrera #1 had reached capacity at the time of the preparation of this study). The maximum elevation for the VBK south rock storage pile is 340 masl. The maximum elevation for the VBK north rock storage pile is 335 masl. The maximum elevation for the VBK west rock storage pile is 340 masl. Additional lifts were added to the Escombrera Buena Vista storage pile for a maximum elevation of 370 masl.

For the Incredible 6 pits, waste rock storage is located immediately south and north of the planned ultimate pits. Maximum elevation of the waste piles is 225 masl for the south rock storage facility and 210 masl for the north storage facility. Maximum vertical lift for each pile is 45 m.

It should be noted that the rock storage facilities have only been sited and designed at a conceptual scale. Additional geotechnical analysis is required to assess the stability of the proposed designs and locations of the dumps.

16.7 ORE STOCKPILES

To minimize risk and to ensure a steady flow of mill feed to the plant, a conservative ore stockpile strategy was developed. In addition to short term run-of-mine stockpiles located near the crusher, longer term ore storage capacity was explored. The main pit for the Pisolita deposit is large and shallow, making it a suitable location for longer term ore stockpile storage. As per current operating practices, ore stockpiles will be sorted into low-grade, medium-grade, and high-grade stockpiles using the cut-off grades of 0.40%, 0.7%, and 1.0% respectively. A closing stockpile balance of 75,000 tonnes is brought forward from 2010.

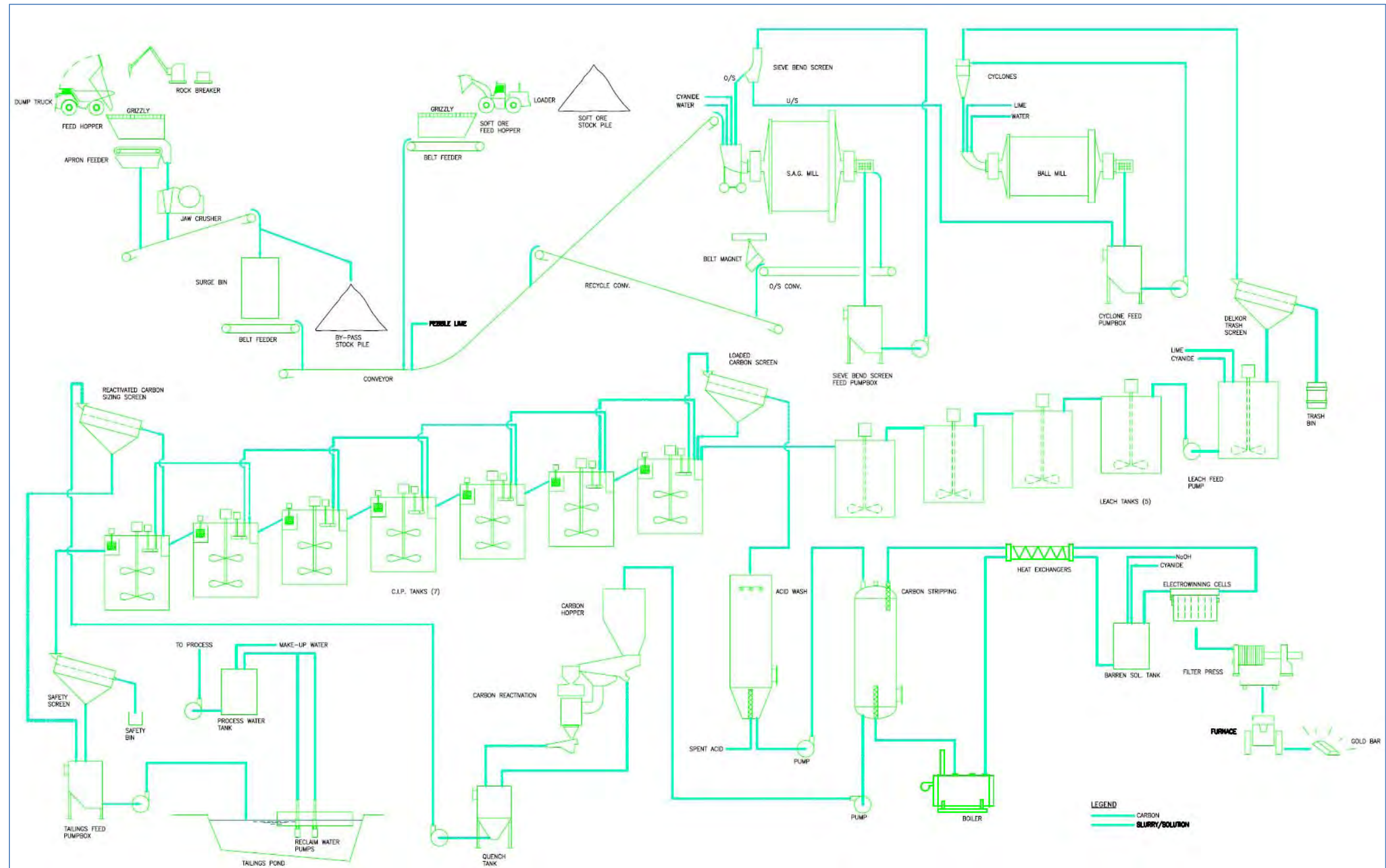
17.0 RECOVERY METHODS

17.1 CURRENT PROCESSING FACILITIES

The Choco 10 gold recovery plant was installed and commissioned in 2005, with the majority of equipment sourced from a previously closed operation in Canada. Under Rusoro's management the plant is reported to have achieved a throughput of close to 7,000 t/d while treating saprolitic material. For the purpose of this study, however, a throughput of only 5,000 t/d has been assumed, well within its rated capacity of 5,400 t/d when treating harder rock.

The process flowsheet is largely conventional, consisting of primary crushing, two-stage milling, cyanide leaching, carbon adsorption and elution, electro-winning and gold smelting. Tailings disposal is undertaken in a valley dam, with decant solution returned as process water to the plant. Figure 17.1 shows a general process flowsheet for the existing Choco 10 gold recovery plant.

Figure 17.1
Schematic Flow Diagram of the Existing Plant (5,000 t/d)



17.1.1 Crushing and Grinding

Run-of-mine ore is dumped directly by truck or front-end loader into a bin with a grizzly and the undersize fed onto an apron feeder. Material from the apron feeder is fed onto a screen where undersized material bypasses the primary jaw crusher. A separate feed bin is also in place to allow for bypassing the crusher when the feed material is predominantly oxide (saprolitic material). The crusher product, at approximately 80% minus 6 in, and undersize material are fed directly onto the SAG mill conveyor feed-belt. Lime is added onto the SAG mill feed-belt via a bin and screw conveyor.

Milling is undertaken in a 20 ft (6.1 m) diameter by 17 ft (5.2 m) long SAG mill (2,750 HP : 2,052 kW) operated in either open or closed circuit with the SAG mill product feeding a 17 ft (5.2 m) diameter by 23 ft (7.0 m) long ball mill (2,750 HP : 2,052 kW). A facility is in place to return pebbles back to the SAG mill feed-belt. The ball mill product feeds a cyclone pack and the cyclone overflow is screened on a 12 m² Delkor linear screen for removal of trash. The ball mill operates in closed circuit, with cyclones.

17.1.2 CIP Leaching and Gold Recovery

Cyanide leaching of the milled product is undertaken in five large leach vessels each of 3,450 m³. The leach vessels are mechanically agitated with air injection to maintain dissolved oxygen at an acceptable level. The installed leaching capacity allows for theoretical leach residence time in excess of 48 h.

The slurry is transferred from the leach tanks to a conventional carbon-in-pulp circuit, comprising seven CIP tanks, operating in a conventional counter-current flow of slurry and carbon. Carbon is transferred upstream by recessed impellor pumps with inter-stage screens, recently upgraded from Kambalda screens to two 12 m² Kemix screens. Loaded carbon is pumped from the head tank to the elution and regeneration section for stripping of contained gold. The regenerated carbon is returned to the tail adsorption tank.

Elution of loaded carbon is carried out in a 4 t batch through a conventional Pressure Zadra vertical column. Acid washing of the loaded carbon is undertaken predominantly to reduce calcium loadings on the carbon. Carbon regeneration is carried out in a vertical kiln with typical temperatures of 700°C. Tails from the adsorption circuit are screened for removal of carbon, and pumped directly to the tailings dam.

17.1.3 Sampling Analysis, Gold Accounting and Security

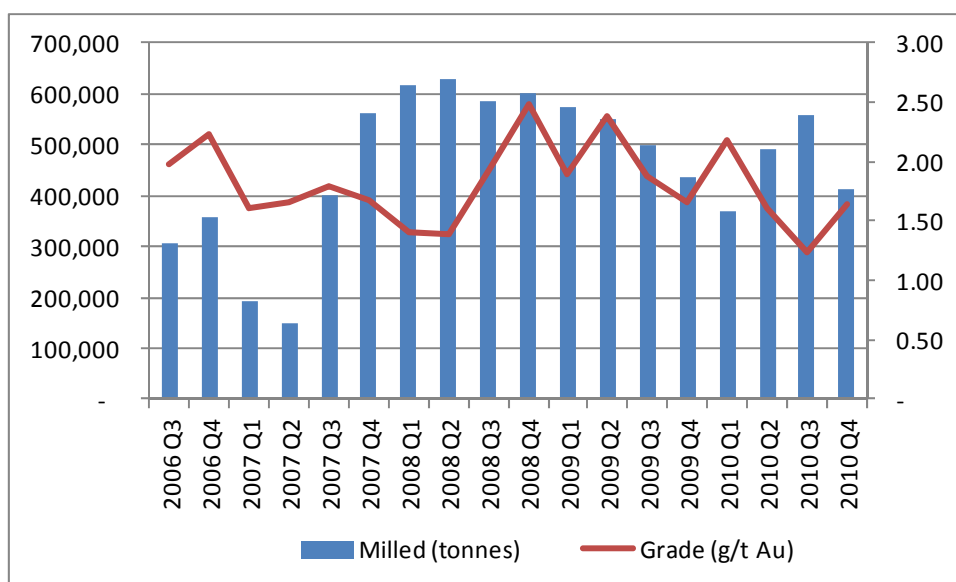
Plant feed tonnage is generally measured by weigh scales on the mill feed belts. Capital has been expended to upgrade methods to determine the metallurgical plant balance. Upgrade of plant security is in progress, including installation of closed-circuit security cameras and construction of a secondary security fence around the plant to better control access to the facility.

17.1.4 Production

The plant is designed to operate as a SAG/ball mill operation; however, the current feed is predominantly saprolite/oxidized material with very low work indices and requires little energy for grinding. At the time of Micon's site visit (see Leader et al., 2007) milling was set up to operate with only the SAG mill. When operating in this mode, the plant is capable of treating the equivalent of nearly 7,000 t/d of saprolitic ore.

Figure 17.2 illustrates the production history from mid-2006, as reported by Rusoro. These data are for 3 monthly intervals (quarters).

Figure 17.2
Historical Production from the Choco 10 Plant



A review of the various reports relating to operating costs for the plant indicates that these are in reasonable agreement with those to be expected for a similar plant treating this type of ore. Additionally, the reagent consumptions also agree well with those on which the feasibility study is based. Barring external factors causing disruption to normal operations, as operating time improves unit operating costs should also reduce. However, these costs will increase somewhat when harder, fresh ore is treated.

17.2 PROPOSED EXPANSION OPTIONS CONSIDERED

The intent of the increase in plant capacity is for the Choco 10 plant to be able to produce up to 500,000 oz/y of gold utilizing a flowsheet that is essentially similar to the existing one. In the 2009 Preliminary Assessment (Buchanan et al., 2009), a number of plant scenarios were considered:

- Expand the existing 5,000 t/d plant.
- A new 5,000 t/d plant to operate in parallel with the existing facility (total 10,000 t/d).
- A new 10,000 t/d plant (and shut down the existing plant).
- Existing 5,000 t/d plant plus a new 10,000 t/d plant (total 15,000 t/d).
- Existing 5,000 t/d plant plus a new 15,000 t/d plant (total 20,000 t/d).
- A new 20,000 t/d plant comprising two 10,000 t/d grinding lines and shut down the existing plant.

The economic analysis presented in that study demonstrated that keeping the existing mill at 5,000 t/d and adding a new, 15,000 t/d plant provides the optimum return on capital. Thus, the feasibility study envisages the existing plant continuing to operate “as is”, with the crushing, grinding and leach circuits as shown in Figure 17.1, above.

Details of the proposed plant were noted in the 2009 Preliminary Assessment. This information provided the basis for a feasibility study to determine more precisely the economics of the expanded plant.

For the feasibility study, the basic engineering and construction capital estimate for the proposed plant expansion and associated infrastructure was completed by Ausenco.

The flowsheet for the combined 5,000 t/d and new 15,000 t/d facilities is shown in Figure 17.3. The layout of the proposed facilities is shown in Figure 17.4.

Figure 17.3
Flowsheet for Expansion to 20,000 t/d (5,000 + 15,000 t/d)

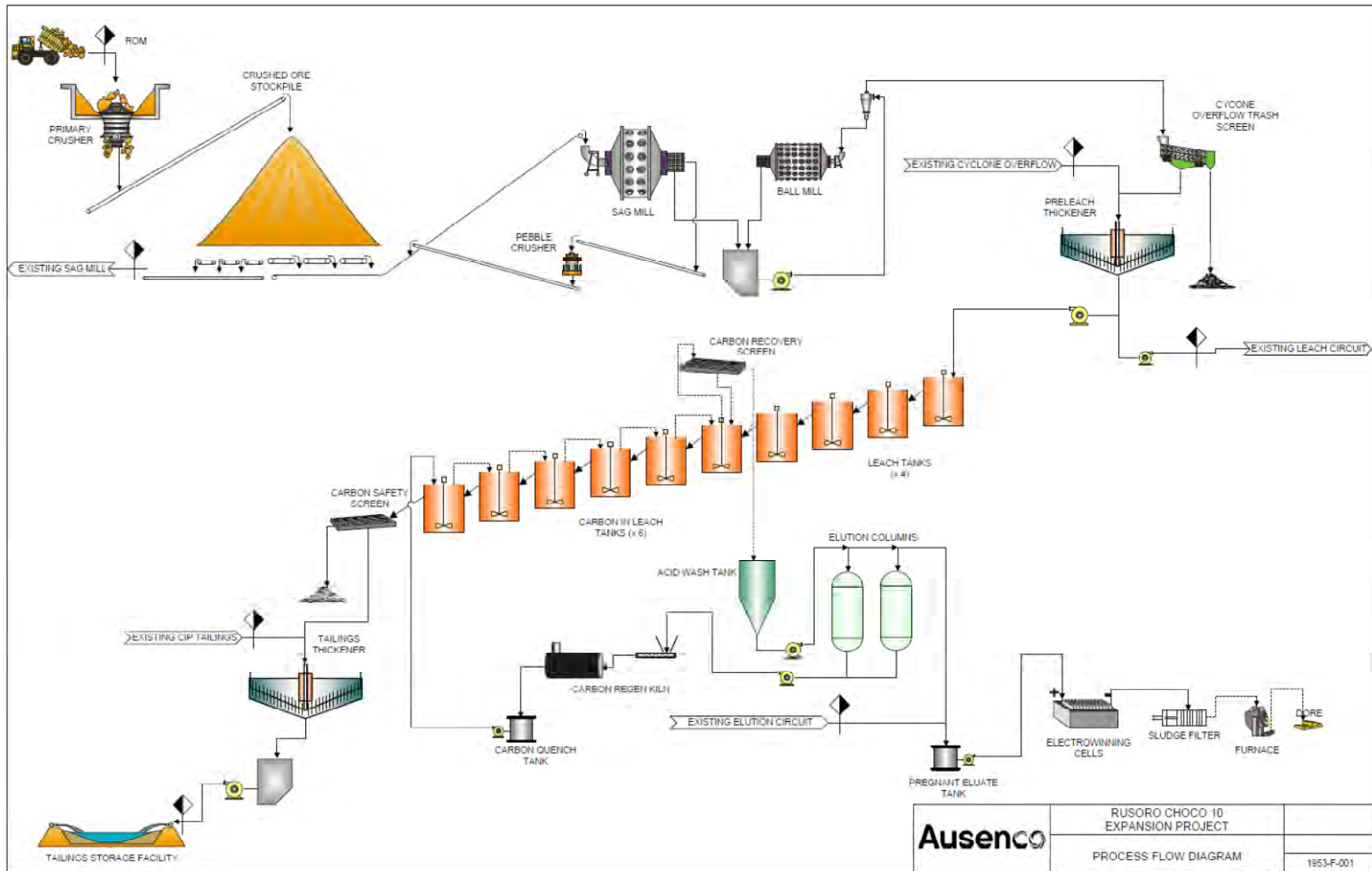
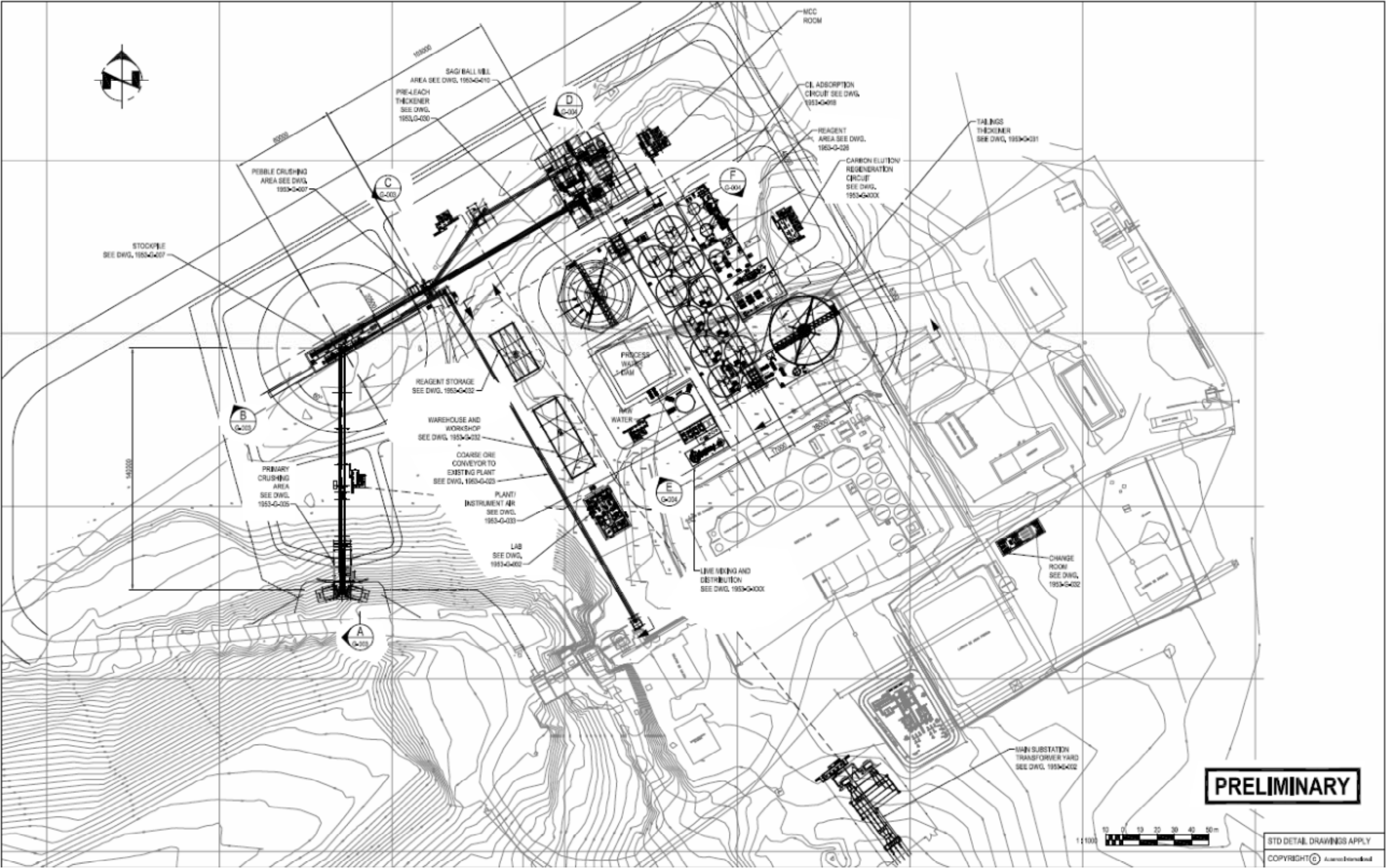


Figure 17.4
Proposed Site Layout
(Source: Ausenco Dwg 1953-G-002)



17.3 PROCESS PLANT DESIGN BASIS AND DESCRIPTION

17.3.1 General

The process plant and associated service facilities will process run-of-mine (ROM) ore delivered to the primary crusher to produce doré bars and tailings. The process encompasses crushing and grinding of the ROM ore, pre-leach thickening, carbon-in-leach (CIL) gold extraction, carbon stripping, electrowinning, and smelting to produce gold bars that will be shipped to a refinery for further processing. The CIL tailings will be thickened before placement in the tailings storage facility (TSF).

The new plant will be integrated with the existing facilities to maximise the plant operability and throughput while minimising capital expenditure. Figure 17.4 (above) shows the current proposed design which permits a simple integration of the expanded facilities into the existing plant and minimises downtime for the “tie-in.”

17.3.2 Process Plant Design Basis

The Feasibility Study (FS) design criteria were based on the available test work as well as data from similar operations. The timeline of the study meant that the scope of the metallurgical testing typically recommended for an FS could not be completed in time for incorporation in the study. Benchmarking against similar operations was therefore required to confirm some of the design criteria. Ausenco considers this approach to be adequate in terms of managing risk for this project at the FS level of accuracy.

The key criteria selected for the plant design are:

- Treatment of an average 15,000 dry metric tonnes per day (t/d) for 365 days per year (d/a), after allowance for availability, while treating 100% primary hard ore.
- Design availability of 90%, assuming 7,884 operating hours per year, with standby equipment in critical areas.
- Sufficient plant design flexibility for treatment of all ore types at design throughput, as indicated by the completed test work.

The selection of these parameters is discussed further below.

17.3.3 Throughput and Availability

An overall plant availability of 90%, or 7,884 h/a, was nominated. Benchmarking indicates that well-operated plants with abrasive ores similar to those tested for this project have consistently achieved 91% overall plant availability.

The selected throughput is mainly a function of the mining production schedule. The recommended rate of 15,000 t/d through the new plant to allow a total operational throughput of 20,000 t/d, is based on the hard rock capacity of 5,000 t/d for the existing operation.

17.3.4 Processing Strategy

The processing strategy is designed to maximize the total facility throughput by incorporating existing plant unit processes where practical. For example, the new gyratory crusher and coarse ore stockpile (COS) will feed both the existing and new milling circuits. The existing crushing circuit will be retained as feed source for the to the existing plant, allowing the two plants to be fed separately and the existing crushing plant to operate when the gyratory crusher is down for maintenance.

The two milling circuits will operate independently through to the classification cyclones. The cyclone overflow from each circuit will be combined in the new pre-leach thickener. Underflow from the thickener will be divided and pumped separately to the new and existing leach tanks. CIL/CIP tailings from the respective circuits will be combined again in the tailings thickener.

Loaded carbon from each circuit will be stripped in separate (new and existing) elution circuits. The acid wash and elution columns for the existing plant will remain in their current locations, while all electrowinning and gold handling will be consolidated into a new and common gold room.

17.3.4.1 Head Grade

The plant is designed to treat various tonnages of ore at a maximum feed rate of 15,000 t/d and a grade of 2.5 g/t Au.

17.3.4.2 Process Plant Design Criteria Summary

The overall design approach is to provide a robust process plant that can handle the variability in the metallurgical performance of material from the Choco 10 and Increible 6 ore bodies, as defined during the test work programs. The design of the new plant focuses specifically on the metallurgical performance of the hard rock samples from the various pits and does not consider the characteristics of the saprolite ores which are expected to be exhausted prior to the expanded plant commencing operation.

A detailed process design criteria derived from the results of the metallurgical test work program has been developed and the key criteria are summarized in Table 17.1.

Table 17.1
Process Design Criteria Summary

Criterion	Description	Units	Design
GENERAL			
Head grades	Gold	g/t	2.5
Crushed ore bulk density		t/m ³	1.60
Moisture in ROM feed	Design	% H ₂ O	3%
Hard Rock Ore			
Crushing Work Index	Design	kWh/t	23.9
Unconfined Compressive Strength	80 th Percentile	MPa	83
Drop Weight Index	Design	kWh/m ³	8.37
Bond Rod mill work index	Design	kWh/t	18.1
Bond Ball mill work index	Design	kWh/t	14.5
Abrasion index	Design	g	0.59
PLANT OPERATING SUMMARY			
Annual ore treatment		t/a	5,475,000
Operating days per year		d/a	365
Available hours per day		h/d	24.0
Plant availability		%	90.0
Design feed rate		t/h	694
Operating hours per year		h	7,884
Nominal plant throughput per day		t/d	15,000
Carbon In Leach (CIL)			
CIL dissolution	Au	%	91.0
CIL carbon adsorption	Au	%	99.0
Overall recovery	Au	%	90.1
COMMINUTION			
Primary crusher:	Type		50" x 65" Gyratory
	Crusher Product P80	mm	120
	OSS	mm	165
SAG Mill:	Speed control		Variable - SER Hyperdrive
	Drive System		Twin Pinion
	Dimensions	m	9.75 x 4.26
	Installed Power	kW	8,000
Pebble Recycle Crusher:	Type		HP 400 Short Head Cone
	Design Specific Energy	kWh/t	1
Ball Mill:	Type		Overflow
	Mill speed, % of critical		76%
	Drive System		Twin Pinion
	Dimensions	m	7.16 x 11.28
	Installed Power	kW	11,000

Criterion	Description	Units	Design
Cyclones:	Circulating load		350%
	Overflow P ₈₀	µm	75
	Cyclone Diameter	mm	400
	Cyclone overflow density	% w/w	35
PRE-LEACH THICKENER			
General:			
	Thickener type		High rate
	Flocculant addition	g/t	10
	Settling rate - range	t/m ² h	0.75
Underflow density:			
	Design	% w/w	55%
CARBON IN LEACH			
Arrangement			
	Number of leach tanks		4
	Number of CIL tanks		6
	Total residence time	h	48
Leaching			
	Gold dissolution		91%
	Tailings solution grade gold	Au g/m ³	0.05
Carbon Parameters			
General:			
	Type		Coconut shell
Metal grades on carbon:			
	Loaded carbon - gold (Nominal)	g/t	2,500
Carbon Kinetics - Design			
	Fleming k value		130
	Fleming n value		0.6
TAILINGS THICKENER			
General:			
	Thickener type		High rate
	Flocculant addition	g/t	10
	Settling rate - range	t/m ² h	0.75
Underflow density:			
	Design	% w/w	55%
DESORPTION AND ELECTROWINNING			
Carbon batch size		tonnes	20.0
	Method		Split Zadra
	Elution temperature	°C	130
Electrowinning			
	Type		Sludging

17.3.5 Unit Process Selection

The process plant design is based on a flow sheet incorporating the following unit process operations:

- Crushing ore from the open pits in a primary gyratory to a product size of nominally 80% passing (P80) 120 mm for feed to a coarse ore stockpile (COS)
- COS with a live capacity of 20,000 t (24 h) to provide crusher product surge capacity ahead of SAG milling
- Two coarse ore reclaim systems to feed the existing and new plants
- 8 MW SAG mill in closed circuit with pebble crushing
- 11 MW ball mill in closed circuit with cyclones
- Pre-leach thickening in a high-rate thickener to an underflow density of 55% solids
- 4 x 5,000 m³ live capacity cyanidation leach tanks in series
- 6 x 5,000 m³ live capacity carbon-in-leach (CIL) tanks in series
- 10 tonne carbon acid wash and two 10 tonne carbon elution columns
- 20 tonne carbon regeneration kiln
- Electrowinning via six sludging cells with stainless-steel wool cathodes
- Electrowinning sludge drying oven and diesel-fired crucible smelting furnace
- Tailings thickening in a high-rate thickener to an underflow density of 55% solids
- Centrifugal pumping of tailings to a conventional, tailings storage facility (TSF)
- Raw process plant water supply from the existing site raw water dam; raw water distribution throughout the plant as required
- Process water supply from water reclaimed from the TSF, bore hole wells, mine decant, and process operations
- Sewage treatment plant for larger labour force
- Plant and instrument air services and associated infrastructure

17.3.5.1 Primary Crushing

Based on the design throughput and ore characteristics, a gyratory crusher is considered the most suitable primary crusher for the duty. The smallest available gyratory crusher is 42" x 65" in size. Although this crusher is capable of crushing more than 20,000 t/d, which would allow all ore to be processed through the gyratory facility and bypass the existing crushing circuit, a slightly larger crusher, 50" x 65", was selected based on its ability to accept a larger feed size (~1 m).

The concept for the crushing station is a minimalist design with a low concrete structure, two feeding sides, and no storage bin or apron feeder below the crusher; the design includes a rock box to protect the stockpile feed conveyor belt and room to access and remove the eccentric for maintenance. The design incorporates continuous feeding with 220 tonne trucks and occasional reclaim from the ROM stockpile with a front-end loader.

The selected crusher can handle feed approximately 1 m diameter in size so there will be little need for a rock breaker if the mine can meet this top size criterion. A mobile rock breaker supplied by the mining department will be used for oversize breakage when necessary.

Crusher product discharges directly from the crusher onto the stockpile feed conveyor. A discharge feeder and only a minimal surge bin have been incorporated to minimize the height of the crusher structure. To ensure no build-up of material under the crusher, the stockpile feed conveyor is designed to run at 3,000 t/h, faster than the expected instantaneous fall-through rate possible from the crusher.

The stockpile feed conveyor is equipped with a belt scale and a metal detector.

17.3.5.2 Crushed Ore Stockpile Capacity

Crushed ore will be conveyed to a crushed ore stockpile (COS) that provides 24 hours live surge capacity for both the new and existing plants (20,000 t). Two parallel trains of three apron feeders per train will reclaim ore from under the stockpile to feed the new and existing SAG mills. Each feeder has been sized for 60% of the design mill feed tonnage to allow two feeders in each train to operate with some surge/turndown allowance. Feed to the existing mill will be via the existing SAG mill emergency feeder and from there to the SAG mill feed belt.

During crusher shutdowns, ore can be reclaimed from the COS for both SAG circuits, or the existing primary jaw crusher can be used to feed the existing SAG mill. During primary gyratory crusher shutdowns longer than 24 hours, the dead zone of the COS will need to be "dozed in" to provide ore for the SAG mills.

The stockpile feed conveyor and reclaim tunnels are on an obtuse angle to each other, this has sometimes resulted in a bias in the mill feed sizes for a two train reclaim system, with the

outside mill preferentially being fed coarser material. This can result in a slight reduction in throughput for the outside train and an increase in the inside train. The outside circuit is the larger of the two and incorporated pebble crushing and is expected to be least influenced by any feed size bias.

No cognisance of this effect has been made in the mill design and layout, further work to straighten the angle and minimise this effect may be warranted as part of future work.

17.3.5.3 Comminution Circuit

Mill Power Requirements

Test work data indicate that material from most of the rock types is of moderate to high competency and requires a relatively fine primary grind for mineral liberation. A two-stage milling circuit with pebble crushing (SABC) has been selected as the most practical flow sheet for the required throughput of ore with this competency and grain size.

The major comminution design parameters were based on results from the PFS metallurgical test work program. Additional test work was undertaken during the FS, but the results were not available in time to incorporate into the FS design. A preliminary evaluation of the additional test work results against the FS design, identified minor discrepancies in the installed powers and mill sizings, but was considered too minor to warrant any changes.

The tonnage percentages of the ore resources used for the weighted average for the 12 years of operation are shown in Table 17.2.

Table 17.2
Ore Deposit Resource Tonnage Ratio

Pit	%
VBK	76
Rosika	9
Culebra	4
Elisa	7

The above shows that the VBK deposit forms the major part of the resource although this will likely lessen as the Incredible deposit is explored in more detail. However, the high initial proportion indicates that this rock type is a major consideration in selecting the comminution parameters. The results for the primary hard ores are shown in Table 17.3.

Table 17.3
Primary Ore Comminution Parameters

Sample Name	Location	JKDWT A*b	SMC DWI (kWh/m3)	Bond CWI	Bond RWI	Bond BWI	Abrasion Index Ai
MET VBK 01 D4IHG	VBK		7.05			12.4	0.18
MET VBK 03 FWMIN	VBK		9.92			12.6	0.14
MET ROS 01 MLODE	Rosika		6.9			16	0.67
METCOA 01 MLODE	Coacia		8.37			14.5	0.59
Micon Composite A	VBK		8.45		19.2	12.4	0.36
Micon Composite B	VBK		7.86		19.4	11	0.15
Micon Composite C	Rosika/Coacia	38.34	7.63		19.9	12.9	0.38
Micon Composite D	Coacia		10.1		22.8	14.2	0.47
Incredible Comp 1 2010	Cristina		6.64		17.5	14.2	0.25
Incredible Comp 2 2010	Elisa/Ingrid		8.33		22.6	16.6	0.54
Incredible Comp 3 2010	Elisa/Ingrid		8		20.8		0.22
Incredible Comp 2 2007	Cristina					16.4	0.29
Incredible Comp 3 2007	Culebra					13.6	
Incredible Comp 5 2007	Elisa					15	0.41
Incredible Comp 6 2007	Ingrid					15.5	
CH04	Choco 10					13.6	
CH05	Rosika				17.8	14.1	
CH06	Rosika				16.7	13.7	
Number of tests		1	11	0	9	17	13
Design			8.37	23.9	18.1	14.5	0.59

Various consultants use several techniques to determine comminution power requirements. These methods are typically power-based (empirical modification of Bond power) or model-based, such as JKSimMet modelling.

Ausenco uses a power approach, based on empirically derived models developed from a database of actual plant operating data and associated bench-scale test work. Critical input parameters to the model are ore competency (measured by either JK Drop Weight Axb or SMC DWI values) and Bond work indices (CWI, RWI, and BWI). Ausenco's power-based model uses the JK Drop Weight/SMC data, which are a measure of ore competency, to predict the milling efficiency of the various circuits. The specific energy and mill sizing determined from Ausenco's in-house modelling method are shown in Table 17.4.

Table 17.4
Grinding Mill Design Criteria

Criteria		Units	Design Ore
Throughput		t/h	694
Primary Mill Type			SAG Grate D/C
Pinion Power required		kW	6560
Mill Speed		% Nc	65 - 80
Ball Charge Volume	Nominal, operating	% vol	12
	Maximum for design	% vol	18
Total Charge Volume	Nominal, operating	% vol	26
	Maximum for design	% vol	30
Mill Diameter	Inside shell	m	9.75
Mill Length	EGL	m	4.26
Installed Motor power		kW	8000
Secondary Mill Type			Ball
Grind Size	P ₈₀	µm	75
Pinion Power required		kW	9701
Mill Speed		% Nc	76
Ball Charge Volume	Nominal, operating	% vol	28
	Maximum for design	% vol	33
Mill Diameter	Inside shell	m	7.16
Mill Length	EGL	m	11.46
Installed Motor power		kW	11 000

The installed ball mill power of 11,000 kW incorporates allowances for drive train losses to determine the motor power from the pinion power, as well as a 10% design contingency to account for the accuracy of the models, calculations, and test work used to determine the expected average pinion power. The installed motor power of 8000 kW for the SAG mill incorporates similar allowances plus an additional contingency for adjustments in the mill operating conditions to handle ore variability.

Mill sizing was independently verified by DJB Consultants using the same base criteria which confirmed the suitability of the selected mills for the design duty.

Comminution Circuit Design

Crushed ore is reclaimed from the ore stockpile by separate reclaim trains for the existing and new milling circuits. Each reclaim system consists of three feeders, each equipped with a variable-speed drive, discharging onto a SAG mill feed conveyor. Each reclaim feeder is capable of providing 60% of the design SAG mill feed rate for that train, allowing any two feeders to supply the required mill feed rate with a 20% surge allowance.

The reclaim train feeding the existing plant passes the crushed ore to the existing SAG mill emergency feeder for transfer to the SAG mill feed belt.

The SAG mill feed conveyor for the new plant feeds directly to the new 8 MW SAG mill. A belt scale on the SAG mill feed conveyor monitors the feed rate to permit control of the reclaim feeders. The SAG mill is equipped with a twin-pinion, variable-speed drive system, high-speed motors, and reduction gearboxes, allowing the mill to operating at 65% to 80% of critical speed.

Discharge from the SAG mill gravitates through a trommel and into the mill cyclone feed pump box. Oversized pebbles from the trommel screen (mill rejects/scats) are recycled back onto the mill feed conveyor and reintroduced into the mill after any tramp metal (primarily from ground down mill balls) is magnetically separated out and the pebbles have been crushed to below nominally 12 mm in a pebble crushing circuit.

The ball mill, nominally 7.16 m diameter x 11.46 m EGL, is supplied with cast-steel liners, twin 5.5 MW wound rotor induction motors, a liquid resistance starter, a trommel screen, and a retractable feed spout/chute. Discharge from the ball mill gravitates through a trommel and into the common mill cyclone feed pump box. The combined mill discharge slurry is pumped to the mill cyclone cluster operating in closed circuit configuration with the ball mill. Water is added to the cyclone feed pump box to achieve the required cyclone feed pulp density.

Mill cyclone underflow gravitates to the ball mill feed and the overflow gravitates to the CIL circuit feed, over a vibrating trash screen.

The milling circuit consists of a single cluster of twelve 400 mm cyclones, of which up to ten will be in operation at any one time with two on standby. A pneumatically actuated valve on each cyclone allows for unit isolation. Rubber-lined steel pipes, hoppers, and chutes will be installed throughout the grinding circuit for handling the coarse slurry.

17.3.5.4 Pebble Crushing

The circuit design incorporates pebble crushing and conveyors to return the crushed pebbles to the SAG mill feed. The circuit will consist of a single 230 kW pebble crusher.

Based on typical industry experience with ores of similar competency, a pebble circulating load of 33% of the new feed rate was assumed for the design of the pebble crusher. The FS design allows the pebble crusher to be bypassed when offline for maintenance or when the SAG mill throughput can be maintained by operating without the pebble crusher (in SAB mode) to process more friable ore.

17.3.5.5 Mill Circuit Classification

The classification circuit is designed for a maximum circulating load of 350%, a typical design value widely used within industry for an SABC circuit treating competent ore. The slurry is pumped to a cluster of 12 x 400 mm diameter cyclones. Up to 10 cyclones will be online at any time based on the design plant throughput at 350% circulating load. Fine

cyclone overflow (P_{80} 75 μm) will report to the CIL circuit, while the coarse underflow will report to the ball mill for further grinding. Cyclone overflow will pass over a single-deck vibrating screen for trash removal before reporting to the pre-leach thickener.

17.3.5.6 Pre-Leach Thickening Circuit Sizing

Cyclone overflow from both the existing and new cyclone overflow streams will combine in a single new pre-leach thickener. The underflow will be pumped separately to the new and existing leach circuits. No thickening test work was available for the FS, and the design criteria for the thickener, shown in Table 17.5, were therefore based on standard industry values.

Table 17.5
Pre-Leach Thickening Design Criteria

Criteria	Units	Design Ore
Thickener Type		High Rate
Throughput	t/h	926
Thickener Feed Density	% solids w/w	35
Unit Area Thickening Rate	$\text{m}^2/\text{t}/\text{d}$	0.056
	$\text{t}/\text{m}^2/\text{h}$	0.75
Thickener Diameter	m	40
Flocculant Consumption	g/t	10
Thickener Underflow Density	% solids w/w	55

Subsequent testing indicated that the thickener diameter could be reduced and the target underflow density increased. The potential advantages of these changes should be evaluated prior to final design and construction.

17.3.5.7 Gold Leaching and Adsorption

Design Criteria

The major leaching design parameters used for this study were based on the 2007 metallurgical test work program. No carbon adsorption or equilibrium test work was completed for the FS. The carbon adsorption parameters are based on either industry standards or benchmarked plants operating on similar ores. These design criteria are summarized in Table 17.6.

The CIL tanks will contain a wedge wire carbon inter-tank pump screen and recessed impeller centrifugal carbon transfer pumps to progress carbon in counter-current flow to the leach slurry. An average of 833 kg/h of carbon will be moved through the CIL circuit and stripped per day to maintain the maximum total gold loading of 2,500 g/t. The design carbon inventory in the CIL tanks is 204 tonnes, which results in a carbon dwell time of 245 hours.

Table 17.6
Leaching and Adsorption Design Criteria

Criteria		Units	Design Ore
Leaching Circuit Type			Leach - CIL
Throughput		t/h	694
Leach Feed Density		% solids	50
Leach Feed Volumetric Throughput		m ³ /h	814
Total Leaching & CIL Residence Time		h	48
Gold Dissolution		%	91
Carbon Concentration in CIL	Nominal	g/l	8
	Maximum	g/l	15
Carbon Kinetics - Design	Fleming k value		130
	Fleming n value		0.6
Loaded Carbon (gold)		g/t	2,500
Tailings Solution Grade (gold)		Au g/m ³	0.05
Carbon Stripped per Day		kg	20,000

Process Design

Mill cyclone overflow gravitates to a vibrating trash screen to prevent trash from the pit (typically detonation cord, polyethylene pipe, and organic matter) from entering the leaching tanks. Cyclone overflow, pumped from the existing plant, is also fed to the trash screen. The screened-off material (nominally +0.8 mm) is removed and directed to a skip at ground level. Trash screen underflow gravitates through a metallurgical sampler to the pre-leach thickener. Thickener overflow flows by gravity to the process water dam, and two thickener underflow streams are pumped to each of the new and existing leaching/adsorption circuits.

Leaching of precious metals by cyanide occurs in a “hybrid” carbon-in-leach (CIL) circuit consisting of four leach tanks followed by six CIL tanks where metal leaching continues while being adsorbed onto activated carbon. Leached slurry overflows from the leach tanks through the CIL tanks. The tanks are arranged so that any tank can be by-passed for maintenance without disrupting operations.

A hybrid CIL design was selected over the CIP circuit used in the existing plant because its lower capital cost and greater operational flexibility. The hybrid CIL design maximizes carbon contact time, resulting in flexibility and design contingency to allow for the anticipated variations in gold carbon loading kinetics indicated in the test work and as experienced at similar plants benchmarked by Ausenco.

Sodium cyanide solution is added to the leaching circuit through a pressurized ring main. A sparging system injects oxygen into each tank to assist with maintaining sufficient dissolved oxygen in the slurry for efficient leaching.

Each CIL tank is equipped with a twin-blade agitator to ensure uniform mixing and two mechanically driven intertank pump screens to retain the carbon. All tanks are fitted with bypass launder facilities to allow any tank to be removed from service for agitator and tank maintenance. A CIL gantry crane is available to assist in removing intertank screens and tank agitators for maintenance and routine cleaning.

Regenerated carbon from the carbon regeneration circuit is dewatered by passing over the 0.8 x 10 mm aperture barren carbon dewatering screen. The recovered water gravitates to the carbon safety screen for disposal to tailings. The dewatered activated carbon is fed to CIL tank No. 6.

Carbon flows counter-currently to slurry flow from CIL tank No.6 to tank No.1 using recessed-impeller, vertical spindle transfer pumps. Carbon loaded with precious metals (loaded carbon) is transferred from CIL tank No. 1 to the 0.8 x 10 mm aperture loaded carbon recovery screen mounted above the acid wash column in the desorption circuit. The screen underflow gravitates back to CIL tank No.1, and the loaded carbon reports to the screen oversize and subsequently the acid wash column.

Leach tailings discharged from CIL tank No. 6 gravitate to the tailings thickener via the 0.8 x 10 mm slotted aperture carbon safety screen. Any leaked carbon from holes in an intertank screen is captured at the screen oversize, collected in a bulk bag, and manually returned to the CIL circuit. Undersize from the carbon safety screen gravitates through a final tailings sampler to the tailings thickener. Tailings from the existing plant bypass the carbon safety screen and final tailings sampler and feed directly to the tailings thickener.

17.3.5.8 Carbon Desorption and Electrowinning

Summit Valley Equipment and Engineering (SVEE) were commissioned to provide a complete carbon desorption and electrowinning package design for the FS. Details of the scope of supply and equipment selection by SVEE are summarized below:

- A single 10 tonne carbon capacity, fibre-reinforced plastic (FRP) acid wash column will be provided. Loaded carbon will be washed with 3% hydrochloric acid solution in this column to remove acid-digestible foulants.
- The elution circuit will include two 10 tonne carbon capacity stainless-steel elution columns and will use the split Zadra process patented by SVEE. Gold will be stripped off the loaded carbon by a strip solution containing 1% sodium hydroxide and 0.3% cyanide heated to 130°C under pressure.
- Gold will be removed from the pregnant elution solution by electrowinning. Six electrowinning cells will be installed, each with a 2,000 A rectifier. Sludge, containing the electrowon gold will be washed from the cells and filtered in a plate-and-frame pressure filter.
- Filtered gold sludge will be dried in an oven before being smelted in a 125 kW induction furnace.

- Barren carbon from the elution circuit will be re-activated in a 1200 kW electric carbon regeneration kiln at 650 to 750°C.
- Fine carbon will be recovered from various transfer water streams around the desorption circuit via a plate-and-frame pressure filter. The recovered fine carbon will be stored in bulk bags for subsequent treatment off site.

All unit processes and equipment in the carbon desorption, carbon regeneration, electrowinning, and smelting circuit will be capable of treating 20 t/d of loaded carbon with an average gold loading of 2,500 g/t.

Loaded carbon from the CIL tanks is recovered on the loaded carbon recovery screen and directed to a rubber-lined acid wash column with capacity for 20 tonnes. The carbon is washed with raw water to remove any further entrained solids before being washed with dilute HCl. Concentrated acid (HCl at 33% w/w) is added to raw water in the loaded carbon wash tank to provide the required acid wash solution concentration of 3% w/v HCl. The acid wash solution is pumped through the acid wash column at two bed volumes (BVs) per hour. One bed volume equates to 42.6 m³. Acid-soluble foulants (mainly CaCO₃) that have loaded onto the carbon are dissolved by the acid during this wash period. The spent acid wash solution is discharged to the tailings discharge hopper.

Following acid solution contact, the carbon is rinsed with raw water to remove residual acid and to neutralize the carbon slurry. Carbon from the acid wash column is then hydraulically transferred to one of two 10 tonne (carbon capacity) stainless-steel elution columns. Strip solution is mixed to a concentration of approximately 1% NaOH and 0.1% to 0.3 % NaCN and then heated under pressure to 130°C. The solution is pumped in ascension through a vertical bed of carbon residing in the elution column and discharges at the top of the vessel. As the solution contacts the gold-cyanide laden carbon, the gold is desorbed from the carbon and into the strip solution. The Split Zadra system supplied by SVEE was selected for the FS because it requires less time than other methods to conduct back-to-back strips and/or sequential strips of smaller carbon batches.

Following the strip cycle, gold in the pregnant solution is recovered downstream by electrowinning. Precious metals are periodically harvested from the stainless-steel cathodes by manual high-pressure washing the cathodes to produce a gold sludge. The recovered gold-bearing sludge is filtered through a plate-and-frame pressure filter.

The filtrate is manually loaded into a vendor-supplied drying oven and is then combined with fluxes (silica, nitre, borax) and smelted in an electric induction furnace before being poured into doré bars. The gold/silver doré solidifies and is quenched in water, cleaned to remove slag, stamped for identification, sampled for analysis, weighed, and stored in a vault.

A sump in the gold room collects all gold room spillage and is cleaned periodically to remove solid trash. An overflow weir allows liquid spillage to overflow to the gold room sump pump for pumping back to the leach circuit.

Barren carbon from the elution columns is hydraulically transferred to the kiln dewatering screen, the screened carbon oversize is fed to the kiln feed hopper, and the screen undersize, mostly water, gravitates to the fine carbon filter feed tank. The oversize carbon is fed into the vendor-supplied carbon regeneration kiln.

The carbon is re-activated in the kiln at temperatures of 650° to 750°C. Regenerated carbon exits the kiln and is quenched with water in the carbon quench tank. The carbon is then pumped to a barren carbon dewatering screen for reuse in the CIL circuit. The barren carbon screen oversize reports to the activated carbon storage tank before being pumped directly to CIL tank No. 6, while the undersize (mainly water) is directed to the fine carbon filter feed tank.

Fine carbon is pumped from the fine carbon filter feed tank to a plate-and-frame pressure filter. Fine carbon is harvested into bulk bags for subsequent sale and off-site treatment. Filtrate from the fine carbon filter is returned to the carbon transfer water tank.

17.3.5.9 Tailings Thickening and Disposal

The FS design includes a tailings thickener and a nearby tailings storage facility (TSF) for the disposal of thickened tailings and recovery of reclaim water from the TSF surface.

Slurry discharging the CIL circuit passes through the carbon safety screen and gravitates to the tailings thickener, which is used to dewater the CIL tailings to 55% solids prior to discharge.

Thickening test work was not available for the FS design and therefore, the design was based on the same standard industry values as used for the pre-leach thickener. The thickener will treat the combined tailings from both plants and hence will have the same feed rate as the pre-leach thickener. The thickeners will therefore be the same size.

Flocculant is added to the tailings thickener to accelerate the solids settling. The thickened underflow slurry gravitates to the final tailings hopper before being pumped to the tailings storage facility using a two-stage series pumping system. The thickener water overflow gravitates to the tailings thickener overflow tank for re-use as leach feed dilution water. Re-use of cyanide in this overflow recycling process significantly reduces overall cyanide consumption and hence reagent costs.

17.3.5.10 Reagents and Consumables

A number of reagents are used in processing the mineralized rock to produce gold doré.

17.3.5.11 Lime

Lime is used to control the pH in the CIL circuit. Quicklime is delivered in trucks and dumped into a storage shed. The lime will then be slaked in a slaking mill operating with a

degritting cyclone to produce hydrated lime. The slurry will be stored in an agitated tank and distributed to the existing and new plants through separate ring mains. On/off control valves operating in closed loop control with pH meters are used to regulate lime addition.

17.3.5.12 Flocculant

Flocculant is used in the pre-leach and CIL tailings thickeners to enhance the solids settling. One flocculant mixing, storage, and dosing system is considered. Flocculant powder is delivered in 25 kg bags and loaded into a storage hopper. Dry powder flocculant is mixed with raw water to produce a 0.25% w/v solution in the package flocculant mixing system. The mixed flocculant solution is pumped to a storage tank and dosed from there to the thickeners by variable-speed helical rotor pumps. Process water is mixed into the flocculant lines to dilute the solution to 0.025% w/v before being added to the thickener feed slurry.

17.3.5.13 Sodium Hydroxide

Sodium hydroxide (NaOH) is used as part of the elution process to strip precious metals off the loaded carbon. Sodium hydroxide is delivered in 25 kg bags and mixed on site with raw water to a solution strength of 25% w/w. The solution is dosed to the elution circuit by a dedicated variable-speed helical rotor pump. The existing dosing facility will be used to dose NaOH into the existing elution circuit.

17.3.5.14 Sodium Cyanide

Sodium cyanide (NaCN) is primarily used in the leaching process for the dissolution of gold. It is also used in the elution process to help strip precious metals off the loaded carbon.

Sodium cyanide is delivered as dry pellets in bulk bags (1 tonne). The cyanide pellets are loaded into a mixing tank and mixed with raw water into a solution containing 20% w/w cyanide. The solution is then transferred to a storage tank. Cyanide from the storage tank is dosed to both the new and existing leach circuits through a single pressurized ring main. Cyanide solution is also dosed to the elution circuit from the ring main.

17.3.5.15 Hydrochloric Acid

Hydrochloric acid (HCl) is used in the acid wash stage of the carbon stripping circuit to clean the carbon of scale and foulants. It is delivered to site as a concentrate solution containing 33% HCl in 200 litre drums. The solution is pumped from the drums to a storage tank and dosed from there directly to the acid wash circuit by a dedicated variable-speed helical rotor pump. The existing dosing facility will be used to dose HCl dosing into the existing elution circuit.

17.3.5.16 Activated Carbon

Activated carbon is used in the CIL. Precious metals that are leached into solution adsorb onto the activated and recovered to the carbon stripping circuit.

Carbon is delivered in 500 kg bulk sacks and will be added to the carbon attrition tank. The carbon will be lightly agitated to remove fine carbon prior to being pumped to the activated carbon storage tank and subsequently the CIL.

17.3.5.17 Grinding Media

Grinding media is used in the SAG and ball mills as part of the ore comminution process. High carbon steel forged grinding media will be delivered in bulk and stored on site. SAG mill media is added to a metering hopper to feed the balls onto the SAG mill feed belt. Media addition to the ball mill is via a ball bucket and hoist system.

17.3.5.18 Smelting Fluxes

The main fluxes required for smelting the gold doré are silica, sodium nitrate, and borax. These reagents are delivered in 25 kg bags and added directly to the smelting process, as required.

17.3.5.19 Air Services

High-pressure air for plant, instrument air, and oxygen generation requirements will be provided by four rotary-screw air compressors, three serving as duty compressors and one on standby operating in lead-lag mode. All air will be dried by duty/standby refrigerant air dryers. Plant and instrument air will report to dedicated air receivers prior to reticulation throughout the plant.

A take-off from the high-pressure air line (after the dryers) will direct air to a pressure swing adsorption (PSA) oxygen plant, which will generate oxygen for addition to the leach circuit. This oxygen is sparged into the leach and CIL tanks to provide sufficient dissolved oxygen in the leach solution for gold dissolution. An additional line will reticulate oxygen to the existing plant.

17.3.5.20 Water Services

The water services and distribution systems required for the plant include process, raw, fire and potable water services.

Process water is primarily used in the grinding and CIL areas. The source of process water is pre-leach thickener overflow, TSF decant water, and raw water make-up. In addition, a separate water circuit is provided for the tailings thickener overflow, which is used primarily for CIL feed slurry dilution.

Raw water is primarily used for reagent preparation and in the elution circuit. Raw water is sourced from the existing plant raw water dam and will be stored in the new raw water tank before being distributed through the plant.

Fire water is sourced from the plant raw water tank. The lower section of the raw water tank contains a dedicated 4 hour, or 575 m³, fire water reservoir. The fire water reticulation system consists of an electric jockey pump to maintain line pressure, a single electric centrifugal fire water pump, and a standby diesel-powered pump.

Potable water is used through the plant offices and buildings as well as for the safety eye wash and shower facilities. Potable water will be obtained from a new potable water plant.

18.0 PROJECT INFRASTRUCTURE

The additional infrastructure required for the new Choco 10 expansion process plant is outlined below. The existing infrastructure will continue to support the existing 5,000 t/d plant and mine, supplemented with new facilities only where necessary to support the expanded plant. The expansion infrastructure includes an increase in the power supply, fire protection, sewage treatment, water supply and treatment, site drainage, in-plant roads, communication, security, site buildings, and a temporary construction camp facility.

18.1 POWER SUPPLY

The combined power requirements from the new and existing plants is 40 MVA (32 MVA for the new plant, 8 MVA for the existing). This is significantly higher than the capacity of the existing 13.8 kV, 8 MVA line and therefore requires the construction of a new 115 kV overhead transmission line from the existing El Callao substation to a new high-voltage substation at site.

Electricity will then be distributed via 13.8 kV buried power lines, including a feeder to the existing plant HV switchyard for reticulation to the existing operation. All existing plant and infrastructure will be powered through the existing power reticulation system. For the expansion, electricity distribution will include reticulation to all new substations within the plant boundary and new site infrastructure, including warehouses, maintenance workshops, and the mine dry.

18.2 FUEL AND LUBE STORAGE

The site has an existing fuel storage area for diesel to service the mining fleet as well as the medium- and light-duty equipment. Little additional plant mobile equipment will be required for the expanded plant, and the slight increase in demand does not justify additional fuel storage. The mining fleet for the expanded mining operations will require additional fuel storage capacity. However; these requirements are the responsibility of the mining contractor.

18.3 FIRE PROTECTION

The existing fire protection system will need to be expanded to cover the new facilities and will include a fire water distribution system. An underground fire main, consisting of multiple loops, will provide fire water to the various building sprinkler systems, water monitors, and hydrants.

Plant fire water will be supplied from a dedicated reservoir sized for 576 m³ at the bottom of the plant raw water tank. The fire water will be supplied by a pressurized system containing an electric centrifugal pump and a standby diesel-driven pump. A small jockey pump will also be provided to maintain the line pressure.

Fire hydrants, water monitors, hose reel cabinets, wheeled fire extinguishers, hand-held fire extinguishers, and sprinkler systems will be strategically located throughout the site to provide the required fire protection.

18.4 WATER SUPPLY AND TREATMENT

All raw water required for the project will be harvested from the existing borefield, pumped to the existing raw water dam, and pumped to a new raw water tank for distribution around the new plant.

18.4.1 Potable Water

The existing potable water supply is understood to be adequate for the expanded operation.

18.4.2 Sewage Treatment

A vendor-supplied waste water treatment plant (WWTP) will be provided to treat sewage generated from the camp and various site ablution facilities at a rate of 200 m³/d. Sewage will be pumped to the WWTP by transfer pumps installed in enclosed and lined transfer pits at various locations around the site. The WWTP will produce a sterilized solution suitable for re-use within the process plant water system. The sludge produced will be disposed of within the TSF.

18.4.3 Site Drainage

No site-specific drainage study was performed for the FS. The new plant site will be on flat land that previously housed a coreshed and laboratory. Existing drainage works already capture and divert water around the site. It is anticipated that incremental surface runoff at the plant site will be limited, and no detention or holding ponds are addressed in this study. All stormwater run-off from the new plant site will flow into the existing site drainage system.

18.5 IN-PLANT ROADS

The plant roads identified in the study are limited to those within the boundary limits of expanded plant operation. In addition, the existing haul road to the ROM pad will need to be partially re-routed around the new coarse ore stockpile. Details of the new road requirements are shown on the overall plant General Arrangement (Figure 17.4, above).

18.6 COMMUNICATIONS

Telephones and Internet service will be added to the new control room, warehouses, and gold room; at this stage no expansion of the existing PABX facility has been allowed for. Two-way radios are used for communication within the plant operation.

18.7 SECURITY

The entire processing plant will be fenced. The existing site fencing will be modified to integrate the new plant fencing and allow the current access gate at the southeast end of the plant to be used for both plants. The chain-link fence, topped with razor wire, is intended to keep unauthorized personnel out of the plant area.

The gold room within the plant will be fully secured with a screening checkpoint for persons entering/exiting the room. This location also includes a safe, a metal detector, and camera surveillance for additional security. The gold room will contain ablution facilities to reduce personnel traffic during gold processing.

The new (relocated) laboratory and new workshop and warehouse facilities will be constructed within the plant security fenced area.

18.8 SITE BUILDINGS

18.8.1 Plant Administration Building

A new plant administration building has not been considered for this project. It is assumed that the existing building will be adequate for the administration needs of both plants.

18.8.2 Main Administration Building

A new main administration building has not been considered for this project. It is assumed that the existing building will be adequate for the administration needs of the expanded project. This office building will house General and Administrative, Environmental, and Mining management staff.

18.8.3 Security and Medical Treatment

The existing site fencing and access systems will be modified to combine the low-security areas of the existing and new plants and allow the existing low-security gatehouse at the main access road to the administration office area to be used for access to the combined plants. The gatehouse houses the security guards and provides the principal checkpoint for personnel and vehicles entering the administration and warehouse area.

The existing medical treatment room adjacent to the administration building is assumed to be adequate to service the combined operations. Construction contractors will be responsible for the medical needs of their workers.

18.8.4 Laboratory and Geology Sample Preparation

The existing plant laboratory lies within the design footprint of the new plant and will need to be relocated. A new building, of similar size, has been allowed for at the site shown in the

overall plant General Arrangement (Figure 17.4, above). The equipment in the existing laboratory will be relocated and re-used; no allowance was made for new equipment for the new laboratory.

18.8.5 Process Plant Change-rooms and Mess Hall

A change-room/ablution facility and mess hall will be constructed in the secured plant area. Providing this area for meal breaks will reduce traffic in/out of the plant and therefore reduce security risks.

18.8.6 Process Plant Ancillary Buildings and Facilities

A number of other new buildings have been allowed for within the process plant site, including the gold room, various motor control centres (MCCs), CIL titration room, and plant control rooms.

18.8.7 Warehouse

An additional 200 m² maintenance warehouse and a 400 m² covered reagent store have been allowed for. The warehouse will provide covered storage for small and medium-sized spares used by process and mining operations.

18.8.8 Permanent Camp

The existing permanent camp at El Callao will house senior operations and mining personnel. An additional small temporary construction camp will be provided for construction management and supervision. Accommodation and messing for construction labour will be the responsibility of the individual construction contractors; these costs have been captured in the contractors' rates. No additional camp facilities are expected to be required.

18.9 MINE SITE SURFACE INFRASTRUCTURE

18.9.1.1 Mine Equipment Maintenance Building

The mine equipment maintenance building is a standard structural steel, siding and concrete foundations building normally constructed for similar plant sites. No doors are provided for the maintenance bays. The building includes one maintenance bay for mine service equipment that can also serve as a wash bay and three major equipment maintenance bays to accommodate CAT 793D type mining trucks. The building also includes three maintenance bays for small equipment and space for some offices and a lunchroom on the second floor. The wash bay is equipped with a pressure washing system. The maintenance bays are equipped with a 15-t overhead crane. An oil/water separator is included.

18.9.1.2 Mine Dry

No provision is made for additional space for the mine dry as it is assumed that the existing facilities will be sufficient for the new production rate.

18.9.1.3 Truck Scale

No provision is made for modification/addition to the existing truck scale.

18.9.1.4 Plant Mobile Equipment

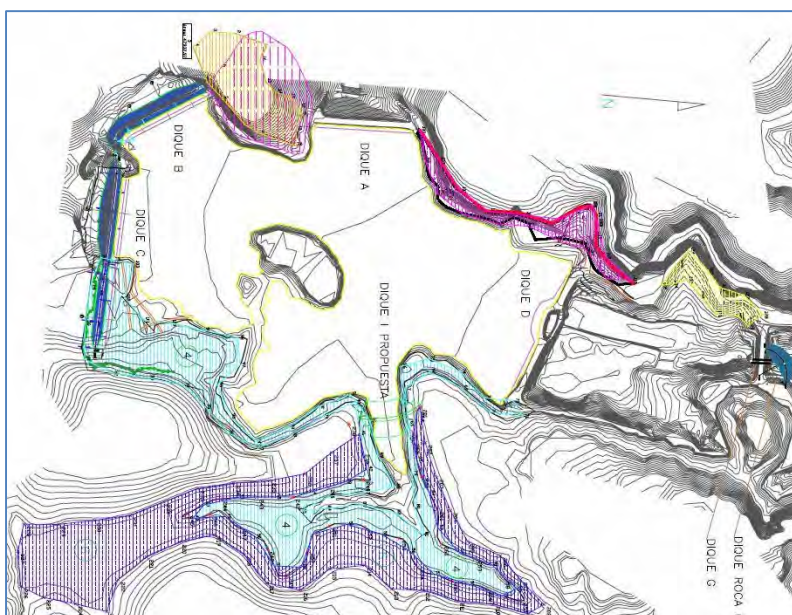
Provision has been made for a 100-t mobile crane to service the new processing facilities. It is assumed that no additional requirement is necessary in terms of plant mobile equipment (pick-ups, small loader, fork lifts, ambulance, fire truck, et cetera) and that the existing plant mobile equipment will cover the new production rate.

18.10 TAILINGS AND WASTE ROCK MANAGEMENT

18.10.1 Existing Tailings Impoundment

The existing TSF site, shown in Figure 18.1 and Figure 18.2, is located approximately 3 km north of the Choco 10 mill, and is currently being used for storage of all tailings. The capacity of the existing TSF is limited by a high voltage electrical transmission line that traverses the facility. PMG surveys suggest that, at the end of 2009, the crest of dams A, B, and C supporting the current TSF were at approximately 225 m and dam D was at 223 m.

Figure 18.1
Schematic Diagram of Existing TSF Layout (not to scale)



Micon understands that, in 2010, PMG prepared to raise the dams as necessary increase the capacity of the existing TSF. The project was designed to increase the capacity of the TSF to approximately 4,000,000 m³ of tailings and provide additional freeboard of 1,850,000 m³ to accommodate any exceptional rainfall events, by raising the crest levels to the 230 m elevation. During 2010, dams A, B, and C were raised to the 230 m elevation, and dam D to an average of 227 m. The approximate level of the tailings in December 2010 was 223.61 m.

18.10.2 TSF Expansion Options

A TSF site trade-off study was carried out by Golder Associates (Golder, 2006). The study evaluated and ranked eight potential sites, named Sites A, B, C, D, F, G, H, and I, for storage of 80 Mt of tailings solids. The sites were evaluated for three different tailings management options: dilute slurry tailings, paste tailings and thickened tailings. The sites were ranked using a number of criteria, including:

- Confining embankment volumes.
- Total plan area footprint.
- Final dam crest and tailings surface elevations.
- Distances from the existing process plant.

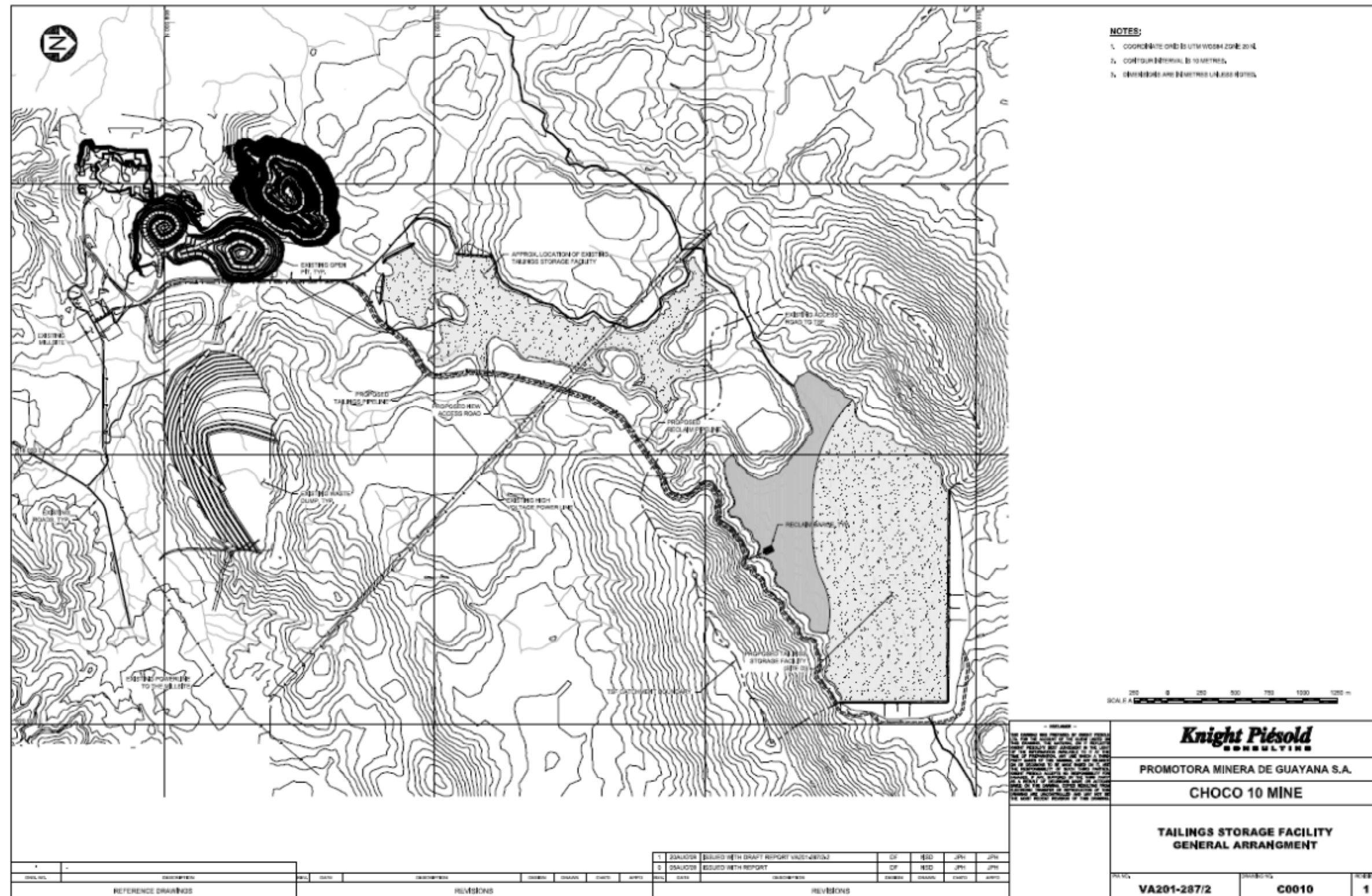
That report recommended that Site D, located in the valley immediately north of the existing TSF be used for the new 70 Mt capacity TSF. Site D was subsequently confirmed as the preferred site by PMG, and is used for the TSF design described herein (Figure 18.2).

18.10.3 Proposed TSF Expansion

Knight Piésold was retained by Rusoro to complete a conceptual design of a tailings storage facility (TSF) at the previously identified Site D, situated approximately 6 km east-northeast of the current process plant, in the adjacent basin immediately northeast of the existing TSF. The principal objectives of the design for the TSF were to ensure protection of regional groundwater and surface waters both during operations and in the long term, and to achieve effective reclamation at mine closure. Features taken into account in the design include:

- Permanent, secure and total confinement of tailings and process water within an engineered tailings management facility
- Control, collection and removal of free draining liquids from the tailings during operations, with recycling, to the maximum practical extent, for process use
- The inclusion of monitoring for all aspects of the facility, to ensure that performance goals are achieved and design criteria and assumptions are met, and
- Staged development of the facility to reduce initial capital expenditures and smooth sustaining capital costs over the life of the project.

Figure 18.2
Proposed TSF Layout (Site ‘D’)



18.10.4 Tailings Characteristics

Three rock types will be processed at the Choco 10 mill and stored in the 70 Mt TSF: oxide, transition, and fresh. The different ore types will result in tailings with distinct characteristics. Test work has been carried out on the oxide and fresh ore tailings to determine the physical and geochemical characteristics. Transition tailings comprise approximately 2% of the total tailings produced over the life of the mine, therefore, test work was not considered necessary. For the purposes of this study, the transition tailings are considered to exhibit similar characteristics to the oxide tailings. Oxides and transition material will together comprise less than 6% of the material stored in the proposed TSF.

18.10.4.1 Oxide Tailings

Golder completed physical testwork on oxide tailings in March 2006 at its laboratory in Sudbury, Ontario. The material characterization program included Particle size analysis (laser scanner and hydrometer), Specific Gravity determination, chemical and mineralogical analysis, paste rheological index testing, paste water retention/pumpability at 7 to 10-inch slump, paste slump and viscosity vs. pulp density relationship, paste yield stress, and dewatering testing (conducted by a vendor).

Rheology test results indicate that the tailings become non-segregating when thickened to a consistency of 34% solids by weight or more.

The geotechnical portion of the testing program was completed by the Golder laboratory in Lakewood, Colorado. The material characterisation program included particle size distribution, Atterberg limits, shrinkage limit, Specific Gravity, slurry consolidation tests, and triaxial compression tests.

18.10.4.2 Hard Rock Tailings

Hard rock tailings will account for approximately 94% of the total tailings produced over the life of the mine. Tailings produced from hard rock samples underwent physical, rheological, and geochemical laboratory testing by Knight Piésold (KP), Paterson and Cooke (P&C), and CANTEST, respectively.

The following points summarise the findings of the tailings tests:

- The tailings sample is described as a non-plastic SILT with SAND.
- The settled dry density of the tailings is approximately 1.24 t/m³ for undrained settling conditions and 1.32 t/m³ for drained settling conditions. Measured supernatant water release was 41% and 43% for the undrained and drained settling tests respectively. The tailings slurry took approximately two days to complete undrained or drained settling.

- Coefficients of consolidation determined for the tailings generally increase with increasing effective confining stress, ranging from about 220 to 2,781 m²/y (for confining stresses ranging up to about 827 kPa). These coefficients of consolidation are within the range of typical values for sandy silt tailings materials and are higher compared to more fine grained tailings.
- Calculated vertical permeabilities for the tailings range from approximately 1.0 x 10⁻⁵ cm/s at very low stresses, decreasing to about 4.8 x 10⁻⁶ cm/s at high stresses.
- The dry density of the tailings increases with increasing effective stress (due to consolidation), with a value of about 1.7 t/m³ achieved at an effective stress of approximately 827 kPa.
- A tailings dry density of approximately 1.47 t/m³ is achieved under air drying conditions.

The slurry consolidation test results indicate that the consolidation and permeability characteristics of the tailings are consistent with other similar hard rock mine tailings (sandy silt) materials.

18.10.5 Site Characteristics

18.10.5.1 Physiographic Setting

The proposed TSF site shown is located in a broad, gently sloping valley basin varying in elevation from approximately 200 m above sea level (masl) at the northeast, to 250 masl at the height of land to the southwest. The valley sides to the northwest and southeast have average slopes between 25 and 50%.

The basin is generally covered by dense savannah type forest approximately 15 m in height, although a portion of the proposed tailings site in the vicinity of the main confining embankment is being used as agricultural land and has been cleared of vegetation.

18.10.5.2 Hazard Classification

A hazard assessment has been carried out to enable appropriate design earthquake and storm events to be determined for the TSF. The selection of appropriate design earthquake and flood events has been based on classification of the tailings dam using criteria provided by the Canadian Dam Association's "Dam Safety Guidelines" (2007).

The potential for loss of life following a dam failure is likely minor but cannot be discounted; particularly during operations when there will be work activities downstream of the TSF. If failure resulted in the release of tailings and/or process water it would have a significant environmental impact on the areas downstream. The economic consequences (including clean-up, repair and remedial works) and socio-economic impact to the mine would also be very high. Consequently, a HIGH consequence category (hazard classification) has been assigned to the TSF.

18.10.5.3 Hydrometeorology

The average annual precipitation at the TSF site was estimated to be 1,094 mm. Monthly mean pan evaporation standard deviation values of 2,070 mm (Caroni Macagua, 1974 to 2007) closely match the average of 1,993 mm per year (Tumeremo, 1972 to 1978). A factor of 0.7 has been applied to estimate lake evaporation values from the measured pan evaporation data. The estimated mean annual lake evaporation at the Choco 10 site is 1,517 mm.

Based on the HIGH hazard classification assigned to the TSF in 18.10.5.2, an appropriate IDF is a probabilistically derived event with a return period of one third between the 1-in-1,000 year and the Probable Maximum Flood (PMF).

Considering the steep catchment area slopes, the lush vegetation, and the relatively impermeable clay and silt surficial soils, a runoff coefficient of 0.75 was assumed during average conditions, increasing to 0.9 during the design storm event due to saturation of the soil.

The design of the TSF includes sufficient capacity and freeboard to store the IDF during operations. The storm storage volume of the TSF that is required and provided during operations is approximately 5.75 million m³, plus an additional 2 m of freeboard.

18.10.5.4 Hydrogeology

Standpipe piezometers were installed in drill holes as part of the geotechnical site investigation program. Groundwater levels were estimated as the holes were drilled and measured a few to several weeks after drilling was terminated to allow groundwater levels to equilibrate. Groundwater levels were found to be between 10 m below to slightly above the overburden-bedrock contact in seven of the nine drill holes with piezometers. One drill hole had a groundwater level approximately 30 m below the overburden-bedrock contact and one drill hole was dry at the depth to which the piezometer was installed. These levels correspond to a depth to groundwater of approximately 30 m along the embankment alignment and approximately 20 m downstream of the embankment alignment.

18.10.5.5 Seismicity

The 1-in-500 year earthquake was adopted as the Operating Basis Earthquake (OBE) for the tailings facility design. For a design operating life of 13 years, the probability of exceedance for the OBE event is less than 3%. The median value of maximum acceleration for the 1-in-500 year earthquake is estimated to be 0.03 g. A design earthquake magnitude of 6.5 is appropriate for the OBE, based on a review of the regional seismicity and the findings of the seismic hazard assessment. The tailings facility would be expected to function in a normal manner after the OBE.

An appropriate Maximum Design Earthquake (MDE) for the tailings facility has been defined based on a dam classification of HIGH (see Section 18.10.5.2). The MDE is the 1-in-2,500 year earthquake, with a median value of maximum acceleration of 0.06g, and a design earthquake magnitude of 7.0. Limited deformation of the tailings dam is acceptable under seismic loading from the MDE, provided that the overall stability and integrity of the facility is maintained and that there is no release of stored tailings or water.

A deterministically derived Maximum Credible Earthquake for the Choco 10 project has been defined as a near-field crustal earthquake (focal depth of 15 km) of magnitude 6.5, resulting in a maximum credible acceleration of 0.44g at the site. The return period of this extreme (hypothetical) event is likely much greater than 10,000 years, based on currently available geological and seismicity information.

18.10.6 Geotechnical Considerations

18.10.6.1 Site Investigations

A geotechnical site investigation of the proposed TSF area was completed between October 2008 and February 2009 that comprised 80 test pits and 11 drill holes. Laboratory testing was carried out on selected test pit and drill hole samples, and Standard Penetration Testing (SPT) and hydrogeological tests were completed at various depths within the drill holes.

Results of test pit and drill hole logging, together with the results of in situ and laboratory testing, allowed for the identification and characterization of geological units within the project area and were used to estimate the geotechnical and hydrogeological parameters of the embankment foundation, impoundment basin, and the proposed embankment fill. The geological units encountered include three overburden soils: residual soils, saprolite, and saprock, and two types of bedrock: intrusive metagabbro (gabbro) and a metalava and metapyroclastic from andacitic to dacitic composition (dacite-andecite) (Golder, 2006).

18.10.6.2 Geotechnical Conditions

The geotechnical conditions within the project area were determined using the results from in situ and laboratory testing of test pit and drill hole samples. Findings include the following:

- The overburden soils are generally of low plasticity.
- The stress strain response of the embankment foundation is generally semi-solid/brittle-solid.
- The embankment foundation is very slightly to slightly compressible and slightly overconsolidated; the embankment fill is slightly compressible. Simplified calculations to estimate the ultimate consolidation due to the weight of the embankment indicate that the foundation may consolidate approximately 1.7 m and that the embankment may consolidate approximately 0.6 m.

- Compaction tests indicate that the maximum density of the embankment fill is approximately 16 kN/m^3 at an optimum moisture content of 21%.
- Based on the information available, the shear strength parameters used for modelling were $\phi' = 30^\circ$ and $c' = 0 \text{ kPa}$ for the embankment foundation and $\phi' = 10^\circ$ and $c' = 100 \text{ kPa}$ for the embankment fill.
- The embankment foundation is medium dense.

18.10.6.3 Hydrogeological Conditions

Hydraulic conductivity values used for modelling the embankment foundation and impoundment basin were: overburden $1 \times 10^{-4} \text{ cm/s}$; bedrock $7 \times 10^{-5} \text{ cm/s}$; and $1 \times 10^{-5} \text{ cm/s}$ for embankment fill.

18.10.6.4 Recommendations for Additional Work

Recommendations following the 2008/2009 site investigation included:

- Collecting and testing additional “undisturbed” overburden samples
- Additional testing of in situ overburden
- Additional testing of bedrock samples, and
- Additional testing of select overburden and bedrock samples to confirm laboratory results.

A number of triaxial tests of “undisturbed” samples were not completed due to insufficient sample size or unacceptable disturbance during sample extrusion. Four Shelby tube (ST) samples were not slated for testing and it is recommended that these samples be considered for triaxial testing. Although a maximum of four ST samples remain to be tested, sample availability, storage conditions and variability in the results may necessitate collection of additional ST samples for triaxial testing. It is also recommended that block samples be collected for direct shear testing.

Another method of determining the in situ overburden shear strength is to perform vane shear tests. It is recommended that vane shear testing be incorporated into any additional geotechnical drilling to provide additional information regarding the overburden in situ shear strength parameters.

The UCS test of a bedrock (gabbro) sample indicated that the tested UCS value was lower than the estimated UCS value. It is recommended that additional UCS testing be completed to confirm this result and, if necessary, adjust the estimated UCS values.

Additional laboratory testing of select overburden and bedrock samples should be conducted at a laboratory with well-known quality assurance and control standards to confirm the results from the GMK and UCAB laboratories. Select duplicate overburden samples were

prepared for this purpose following the site investigation program and were being stored on site. It should be confirmed whether these samples are still available and in what condition they are in.

18.10.7 TSF Design (Site ‘D’)

18.10.7.1 Design Basis

The Choco 10 mill will operate at a proposed total throughput of approximately 20,000 t/d. The Tailings Storage Facility (TSF) proposed for Site D is designed for the complete and permanent secure storage of approximately 70 Mt of tailings, process water, surface runoff, and incident precipitation. Local borrow material will be used to construct the TSF embankments.

For development of the TSF filling schedule, a tailings dry density of 0.67 t/m^3 was assumed during the first year of production, increasing to a final dry density of 1.53 t/m^3 in Year 12. These average tailings dry densities are based on the results of the tailings consolidation analyses.

The facility is designed to continue providing secure storage of water and tailings during hydrologic and seismic events up to and including the Inflow Design Flood (IDF) and Maximum Design Earthquake (MDE).

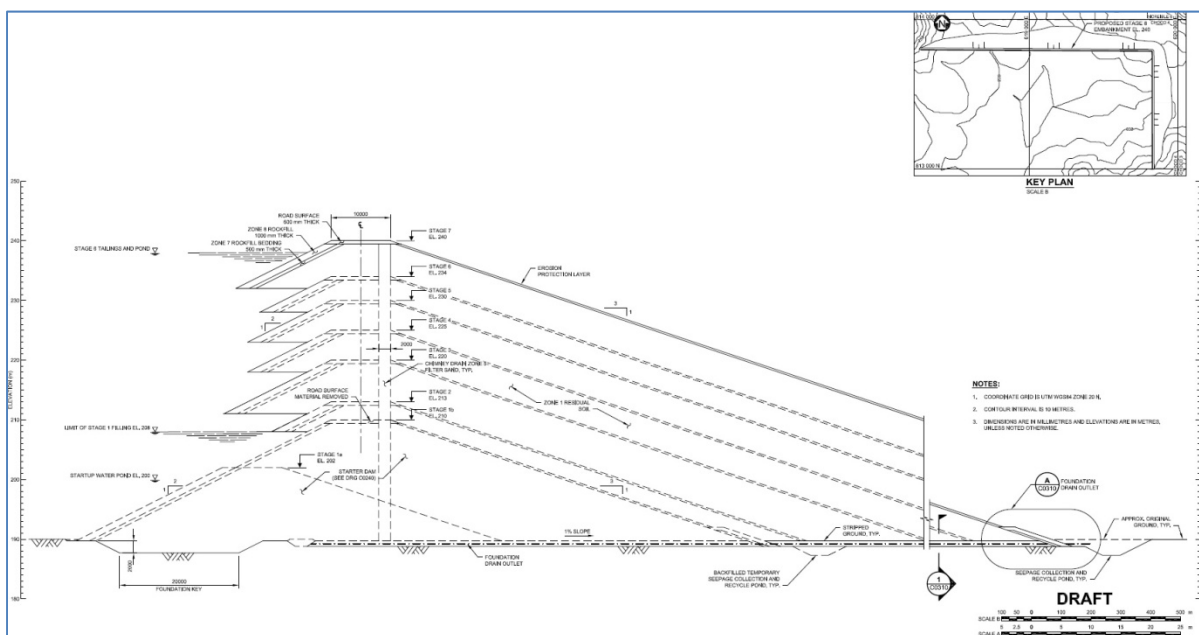
18.10.7.2 Layout and Operating Strategy

The TSF will comprise an earthfill confining embankment along the north and northeast ends of the valley as shown on Figure 18.2. The layout is similar to the Site D dilute slurry tailings option presented by Golder (2006). The dam alignment has been modified to keep the ultimate facility footprint, including the embankment and downstream facilities, entirely within the Choco 4 concession.

The embankment will be developed in stages throughout the life of the project using the centerline construction method. It will be constructed as a water retaining structure with a vertical sand chimney drain running longitudinally along the length of the dam core. The shell zones will be constructed at 3H:1V slopes using low permeability residual soil compacted in lifts. A typical cross section of the final Stage 7 embankment is shown on Figure 18.3.

Design provisions to manage seepage include the in-situ soil liner, vertical chimney drain, foundation drains, and seepage collection and recycle ponds. All seepage losses from the TSF will be collected in seepage collection ponds located at topographic low points downstream of the embankment toe and pumped back to the TSF.

Figure 18.3
Typical Cross-Section of Embankment



Construction of the Stage 1A embankment will start approximately six months before tailings deposition begins in the TSF.

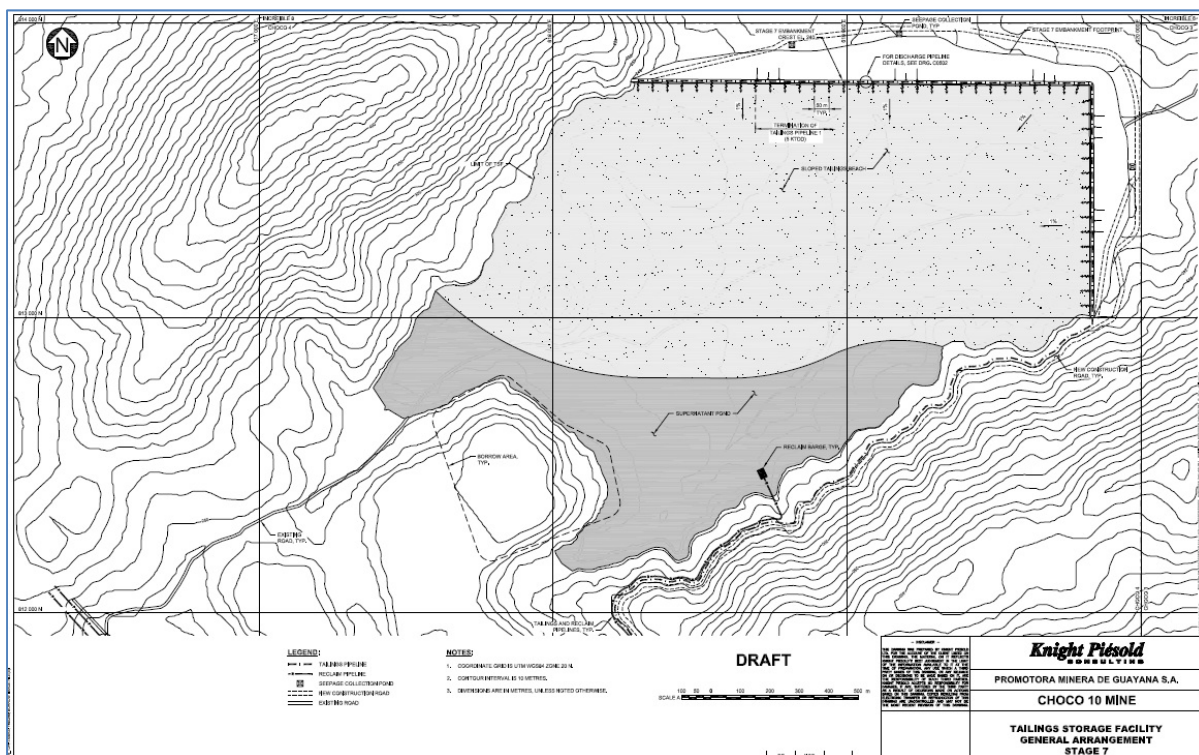
Slurry tailings will be pumped from the mill to the TSF through a High Density Polyethylene (HDPE) pipeline approximately 10 km long. The first 4 km of the pipeline will generally follow an uphill gradient from the mill elevation of 215 masl, to a topographic high point at approximately 280 masl. The last segment of the pipeline will generally be downhill, at approximately 1% to 2% grade, allowing gravity flow to the discharge points at the impoundment.

A multi-stage tailings pumping system located at the mill, designed with full standby redundancy, will provide pressure for tailings transport. A drain-back pond will be located at the mill to facilitate pipeline drainage.

Tailings will be discharged from one of a number of valved offtakes along the dam crest, forming a low-angle beach upstream of the embankment. The systematic rotational deposition of the slurry from the embankment crest will result in the formation of layers of fine, low permeability tailings beaches. Coarser material typically settles rapidly and accumulates closer to the discharge points, forming a beach with a slope of about 0.5% to 1.5%. Finer particles will travel further and settle at flatter slopes. Beach development will be managed to maximise storage volume in the TSF, to control the location of the supernatant pond, and to minimise the potential for fugitive dust emissions. The tailings beach will provide a hydraulically long, low-permeability barrier between the supernatant pond and the dam face. An exposed tailings beach will also promote evaporation from the

tailings, potentially accelerating consolidation. Selective tailings deposition, in coordination with embankment construction, will be used to keep the pond clear of the embankment thereby reducing seepage potential, improving embankment stability, and ensuring that clean reclaimed water is available for reuse at the process plant. A schematic representation of the tailings beach at Stage 7 is shown on Figure 18.4.

Figure 18.4
TSF General Arrangement - Stage 7



18.10.7.3 Seepage Collection and Recycle Systems

Seepage originating in the TSF will be intercepted by the high permeability vertical sand chimney drain within the embankment. Seepage will flow through the continuous sand drain into a series of foundation drains at select low points in the embankment footprint. The foundations drains will be arranged perpendicularly to the embankment centerline, and their outlets will empty into a series of collection ponds excavated downstream of the dam.

Water in the seepage collection ponds will be monitored and recycled to the TSF as required by a system of pumps and pipes.

18.10.7.4 Reclaim Water System

The process water reclaim system must provide sufficient water on a continuous basis from the TSF supernatant pond to support mill operations.

To accommodate pipeline or pump station maintenance, or system breakdown, the ability to deliver water to the mill head tank at a rate greater than the annual average flow requirement is necessary. Additional water supply is available from a water storage reservoir to the west of the mill, and from groundwater wells.

Water from the TSF will be reclaimed from the supernatant pond by pumps mounted on a floating reclaim barge.

Tailings will initially be deposited in the north side of the TSF to allow supernatant water to flow to the south side of the facility. This is closest to the pipeline route back to the mill and the reclaim barge. The reclaim barge will be moored at the south side of the TSF, alongside a causeway or platform constructed from suitable locally excavated material. The causeway will provide maintenance vehicle access, a power supply corridor, and a route for the discharge pipework from the barge. Power will be provided via cable. As required, a skid mounted transformer may be located close to the barge. As the elevation of the supernatant pond rises, access will be maintained by raising the barge access ramps, relocating the transformer, and moving the barge closer to higher ground. Operation of the barge reclaim pump station will be near continuous.

Reclaim water will be pumped from the TSF to the mill through a HDPE pipeline approximately 7 km long. The first 4 km of the pipeline will generally follow an uphill gradient from the reclaim barge elevation of 190 masl to a topographic high point at approximately 270 masl, mirroring the layout of a majority of the tailings pipeline. The last segment of the pipeline will generally be downhill, at approximately 1% to 2% grade, allowing gravity flow to the discharge point at the mill head tank.

18.10.8 Closure

The primary objective of the TSF closure and reclamation initiatives will be to eventually allow the TSF site to become an integrated component of the surrounding ecosystem, mimicking the pre-mining usage and capability of the area. The TSF will be required to maintain long term stability, protect the downstream environment and manage surface water with minimal maintenance.

Activities that will be carried out during at closure to achieve these objectives include:

- Selective discharge of tailings around the facility during the final years of operations to establish a final tailings beach that will facilitate the surface water management and reclamation
- Constructing an engineered cover, including re-vegetation of remaining embankment faces and exposed tailings surfaces using topsoil stockpiled from stripping activities
- Construction of a network of surface runoff swales across the TSF to limit surface erosion and safely convey water to the closure spillway

- Construction of an outlet channel or spillway at the west abutment of the embankment to enable discharge of surface water from the TSF to the downstream watercourse. This full closure scenario will also work well in the event of premature closure of the facility
- Dismantling and removal of the tailings delivery and water reclaim systems and all pipelines, structures and equipment not required beyond mine closure
- Removal of the seepage collection system at such time that suitable water quality for direct release is achieved
- Removal and regrading of all access roads, ponds, ditches, and borrow areas not required beyond mine closure, and
- Long term stabilization of all exposed erodible materials.

Ongoing monitoring will then be required until monitoring results indicate that any seepage from the TSF is of suitable quality for direct release to downstream waters. During the monitoring period, seepage collection ponds and recycle pumps, groundwater monitoring wells and all other geotechnical instrumentation will be retained, the latter for use as long term monitoring devices.

Post closure requirements will also include an annual inspection of the TSF and an ongoing evaluation of water quality, flow rates, and instrumentation records to confirm the design assumptions for closure.

19.0 MARKET STUDIES AND CONTRACTS

19.1 GOLD SALES

As noted in Section 22.0, the forecast sale of gold over the LOM period has been based on a steady gold price of \$1,023/oz, equating to the three-year trailing average spot price over the period to 31 December, 2010.

On July 15, 2010, the Central Bank of Venezuela (CBV) passed Resolution No. 10-07-01 that replaced Resolution No. 09-06-03 and the CBV and Ministry of Finance passed an updated Exchange Agreement No. 12 that replaced the previous version.

Resolution No. 10-07-01 and the updated Exchange Agreement No. 12 became effective August 12, 2010. Resolution No. 10-07-01 mandates that 50% of gold produced in the country in each calendar quarter must be offered for sale to the CBV and after obtaining authorization to export from the CBV, the remaining 50% can be exported or offered for sale to the CBV, at the option of the gold producer. Authorization to export is obtained in the form of renewable permits, which are provided by the CBV and which expire 45 days from issuance.

The updated Exchange Agreement No. 12 mandates that, for companies in which the Venezuelan state has an interest of less than 50%, 50% of proceeds from gold exports collected in a currency other than BsF can be used for direct payments in foreign currency. The remaining 50% of the proceeds from gold exports must be exchanged for BsF with the CBV at the official rate of BsF 4.30/\$1.00. For companies in which the Venezuelan State has an interest of 50% or greater, all proceeds from gold exports collected in a currency other than BsF can be used for direct payments in foreign currency.

Accordingly, this study has assumed that the marketing of production from the Choco 10 and Increible 6 claims is carried out by PMG according to Venezuelan regulations, so that 50% of the bullion is offered to the CBV and the rest exported for sale in international markets.

Micon considers that PMG's continuing access to foreign currency in terms of Exchange Agreement 12 will be necessary to sustain the expanded operation of the Choco 10 mine.

19.2 MATERIAL CONTRACTS

Rusoro contracts out the open pit mining operation at Choco 10, and intends to maintain that contract for the duration of the project. Micon has reviewed the mining contract and is of the opinion that the rates and terms reflect industry norms.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section has been prepared by Jenifer Hill, R.P.Bio., Senior Environmental Scientist with Micon. Ms. Hill is not a Qualified Person in terms of NI 43-101.

20.1 ENVIRONMENTAL STUDIES AND ISSUES

20.1.1 Environmental Conditions

Initial environmental baseline data for the Choco 4 and 10 concessions are included in the 2003 Environmental Impact Assessment, "Mining Development Project Choco 4 and Choco 10", completed by Ambioconsult CA.

The Choco 10 project area falls within the Dry Tropical Forest bioclimatic zone of Venezuela. The weather conditions in this zone are characterized by an annual maximum rainfall pattern, occurring from May to January. Historically, this zone has received an average annual rainfall of over 1,100 mm, with an average maximum rainfall during July of approximately 160 mm and a minimum average of 38.3 mm in March. The mean monthly temperature varies only slightly between 25°C and 27°C throughout the year.

The mine concessions in the El Callao district are entirely in a rural setting. Ambient air quality and visibility are considered good since there are only two small controlled emission points of gaseous and particulate matter in the project area. However, there are some local emissions (e.g., road dust and bush burning).

The majority of the forest in the El Callao district is composed of secondary forest in various stages of development. The area has been extensively disturbed by subsistence agriculture, forestry and artisanal and commercial mining.

Like the flora, the fauna has been extensively disturbed. Hunting is common in the district. This has resulted in a reduced number of larger mammals. During October, 2006 a baseline survey identified 130 different species of mammals, reptiles, fish and birds, none of which is considered rare or uncommon.

The Choco 4 and 10 concessions present a hydrographical dendrite pattern with most of the superficial water intermittent. Both concessions lie inside the Yuraurí River main drainage basin. The only permanent flow of water is the Yuraurí River, which flows from west to east and contains a maximum average of 174 m³/s in August and a minimum average of 2 m³/s in February/March.

The mine is working on progressive reclamation activities which include topsoil spreading on the waste dumps, and revegetation activities through transplanting and raising plants for reclamation in a plant nursery located near the water reservoir.

20.1.2 Environmental Impacts

The Choco property is an operating mine with existing environmental impacts that are being managed according to approved and permitted mitigation measures. The main impacts are the loss of vegetation and wildlife habitat within the footprint of the mine. This is being managed through the concurrent reclamation program. Water is recycled from the tailings facility and there are no surface water discharges from the operation. Groundwater seepages are monitored from the waste rock and tailings facility.

Table 20.1 summarizes the potential impacts and the mitigation for the Incredible 6 operation. These are similar impacts and mitigation identified and managed at the Choco mine operations.

Table 20.1
Incredible Impacts and Mitigation

Potential Impacts	Proposed Mitigation
Open pit development and loss of soils	Hydroseeding to revegetate slopes
Modification to surface runoff and erosion and increased erosion from concentrating surface runoff	Comply with regulation requirements for earthworks
Noise, dust, and other air emissions	Control and mitigation of noise, particulates, smoke and gas contaminants
Loss of vegetation	Fire prevention and fire fighting program
Alteration of water and soil quality by poor material, waste, and effluent management	Adequate management of materials, hazardous and non-hazardous wastes
Wildlife affected by loss of habitat, noise and emissions	Prohibit collection and harassment of wildlife
Alteration of the topography from earthworks	Revegetation of affected areas
Loss of habitat and land use	Restoration and compensation plan
Environmental degradation	Worker environmental awareness and education
Local economic stimulus from contracts	

Note: Adapted from the environmental authorization for exploiting the Incredible 6 deposit (Minamb 2010).

20.1.3 Environmental and Social Management

The focus of the Choco 10 operation has been to ensure legal compliance in all its activities. The operation has an Environmental Management System (EMS) to help manage water, various wastes, hazardous materials, emissions, hydrocarbons, energy consumption, and other environmental protection measures.

The operation is committed to environmentally responsible and sustainable practices and to promote the conservation or enhancement of the natural and social environments in which the company operates. The Choco 10 operation has employed a dedicated Sustainable Development Manager who is responsible for coordinating and communicating with all stakeholders, including communities and the government. The Sustainable Development Manager is also involved with coordinating the community development programs such as construction of a health centre and kindergarten, and road and water supply improvements.

The General Manager is ultimately responsible for environmental management at the Choco 10 mine. The Environmental Manager and the Sustainable Development Manager report directly to the General Manager.

Additional information on the environmental management program is presented in the following sections.

20.2 WASTE AND WATER MANAGEMENT

The majority of mining to date has taken place in saprolite, subsequently the waste rock dumps are not acid generating and seepage is alkaline. Waste characterization studies were completed in 2009 by Knight Piésold and Co. for a variety of rock types including un-weathered rock from below the saprolite. Static testwork indicated that the majority of rock is not acid generating; although there are some samples that were uncertain as to whether they will be acid producing. Leaching tests were also conducted and results showed that there is the potential for waste rock to leach sulphate, aluminum, iron and selenium above Venezuelan guidelines. Water quality monitoring data to date has shown elevated aluminum and iron; however, the high aluminum and iron is indicative of naturally occurring background concentrations associated with the saprolite soils in the area (PMG 2010).

Waste rock dump seepage is monitored, but not collected. A seepage collection system is not planned for future dumps. It is recommended that the operation continues to monitor seepage and make modifications to future dump expansions and new waste rock dumps to allow for a collection and recovery system if seepage quality starts to degrade.

The current tailings impoundment operates as a zero discharge facility. A filter dam is located within the tailings impoundment to allow for water to seep from the tailings into a clear water pond before being reclaimed for re-use in the process plant. A monitoring well is used to monitor seepage quality downstream of the tailings dam. Design provisions to manage seepage for the new tailings impoundment include an in-situ soil liner, vertical chimney drain, foundation drains, and seepage collection and recycle ponds. To the extent possible, seepage losses from the TSF will be collected in ponds located at topographic low points downstream of the embankment toe and pumped back to the TSF.

20.3 PERMITTING REQUIREMENTS

20.3.1 Regulatory Framework

Venezuela has extensive legislation relating to the environment and social protection of the country, including new legislative changes. Key applicable legislation for mine operation is presented in Table 20.2. There is no assurance that future changes in environmental regulation will not adversely affect the operations.

Table 20.2
Applicable Legislation

Legislation	Description
Organic Law of the Environment (updated December 22, 2006)	Establishes inside the national policy the fundamentals to preserve, protect, and improve the environment and a framework for sustainability.
Law of Biodiversity	Governs all the matters related to biodiversity.
Decree N° 1.257. "Norms about evaluation of activities with potential to impact the environment". 13-03-1996	Establishes norms/standards to evaluate activities capable of degrading the environment and also requires the registration of all companies with the Ministry of Environment and Natural Resources (Minamb).
Resolution N° 056. "Norms about environmental evaluation of hydrocarbons and mining projects". 04-07-1996	Establishes the requisites and time periods to apply and obtain permits for mining projects.
Decree N° 883. "Norms for classification and quality control on surface water and water discharge". 11-10-1995	Sets the limits on water discharge and the limits for classification of surface water.
Decree N° 1.400. "Norms about regulation and control of water usage and the watersheds". 10-07-1996	Governs water usage and establishes the requirements to apply for the water usage permits.
Decree N° 638. "Norms about air quality and control of atmospheric emissions". 26-04-1995	Governs and controls air emission and air quality.
Decree N° 2.635. "Norms to control the handling of hazardous material and waste". 22-07-1998	Controls the handling, temporary and final disposal of hazardous material and waste.
Decree N° 2.216. "Norms for handling commercial, industrial, domestic or any kind of non-hazardous waste". 23-04-1992	Governs the handling storage and final disposal of non-hazardous waste.
Decree N° 1.659. "Partial rules of the water and soil woodland law about forest rehabilitation." 05-06-1991 34.808 27/09/1991	Governs the rehabilitation and compensation from the deforestation permits.
Decree N° 3.091. "Technical norms to control the effect associated with gold and diamond mining on the Bolívar state and Antonio Díaz county on Delta Amacuro state"	Addresses at the local level the rules to control the contamination from the gold mining industry.

In order for mining companies to conduct exploration/exploitation activities in Venezuela, the following permits are required:

- Authorization for Occupation of Territory (Occupation Permit). This permit authorises a company's presence at a location, but not the undertaking of any activity. Regulations published under the Decree Law of Mines of 1999 (the Decree Law of Mines), in 2001, require that an Occupation Permit be approved before the granting of mining concessions.
- Authorization for Usage of Natural Resources for Exploration (Exploration Permit). After the occupation permit is granted, a mining company must file an application for an exploration permit with the Ministry of the Environment. The exploration permit must include a brief description of the proposed project, measures for preventing, mitigating and correcting environmental impacts and the conditions and recommendations for the exploration phase. Exploration permits must contain, among others things, the following information: (i) the exploration program; (ii) the quantity of natural resources to be affected by the exploration phase; (iii) identification of impact to the areas to be affected by the exploration; and (iv) a mitigation and recovery plan which must contain plans for the treatment of solid toxic

and non-toxic residues, recovery measures for affected ground, topography and drainage, a reforestation plan and an estimated annual cost. An Environmental Supervision Plan (Supervision Plan) must be filed as part of the Exploration Permit application. The Supervision Plan sets forth the manner in which the execution of the project will be evaluated and controlled.

- Authorization for Usage of Natural Resources for Exploitation (Exploitation Permit). An application for an exploitation permit must be filed with the Ministry of the Environment prior to the commencement of exploitation activities. The exploitation permit must include: (i) a brief description of the proposed project; (ii) a description of plans to prevent, mitigate and correct the environmental impacts of the project; and (iii) the conditions under which the environment may be affected or impacted. A bond issued by a local bank or insurance company must be posted in order to guarantee the execution of the measures necessary for the reclamation of the area and the reduction of the impact of mining activities on the environment during the exploitation phase.

In addition, a number of other authorizations, activities and plans may be required, as indicated below:

- Authorization for water exploitation is required for extracting ground or surface water. Permission is also required for diversion of watercourses.
- Plan for management of hazardous and non-hazardous wastes.
- Environmental Supervision Plan.
- Quarterly monitoring of water discharges and effluents. Air quality monitoring twice a year; rainy and dry season. Yearly check on atmospheric emission.
- Every major change in the original project must be communicated to the Ministry of the Environment, including significant environmental incidents. Some major changes may require an environmental assessment report for Ministry approval, for example new pits, waste dumps or tailings ponds.
- Annual report of hazardous wastes generation, handling and disposal.
- Forestry products resulting from clearing activities must be given to the state through the Ministry of the Environment regional office.
- Various building and construction permits from various regulators.
- Conformance to health and safety standards for the use and storage of cyanide.
- The Ministry of the Environment also requires quarterly environmental supervision reports for the duration of the operations.

20.3.2 Required Environmental Permits

This assessment is for the expansion of an operating mine. The exploitation and exploration permits are in place for Choco 10, Choco 4, and Increible 6, and are kept current. Table 20.3 summarizes the exploitation permits for continued operation and for expansion of the mine operations. Permitting is in progress for expansion of the project to include the new plant, waste rock dumps, tailings impoundment, and ancillary facilities. Certain noted renewals remain outstanding; although, in practice, the expiration date is not applicable once the application for renewal has been submitted.

Table 20.3
Permit Status for Project Changes, New Affection Licences, and/or Exploitation Certificates

No.	Concession	Environmental Authorization	Status	Expiration Date
1	Chocó 4	Authorization for Occupation of Territory	Approved	05-12-2025
2	Chocó 10	Authorization for Occupation of Territory	Approved	05-12-2025
3	Chocó 4	Authorization for Affectation of Natural Resources, for the exploration stage	Renewal Pending	28-01-2009
4	Chocó 10	Authorization for Affectation of Natural Resources, for the exploration stage	Approved	31-03-2009
5	Chocó 4	Authorization for Affectation of Natural Resources, for the exploitation stage	Renewal Pending	28-01-2009
6	Chocó 10	Authorization for Affectation of Natural Resources, for the exploitation stage	Renewal Pending	28-01-2009
7	Chocó 4/ Chocó 10	Activities Accepted for Degrading the Environment, for Gold Exploitation	Approved	No expiry date
8	Increible 6	Authorization for Affectation of Natural Resources, for the exploitation stage	Approved	23-09-2014

A groundwater well field has been permitted and as of the effective date of this report, the Yuruarí River extraction was being permitted for water extraction for the operation for a total of 0.17 m³/s. It was estimated that there will be sufficient water for the expanded operation from these water extraction permits in combination with the existing water reservoir and water recovery from the tailings pond. Additional groundwater well fields may need to be permitted in the future depending on the final water balance.

The Choco 4 exploitation area was expanded to allow for the existing tailings impoundment to operate into 2010. A new tailings facility to be located northeast of the existing facility has been designed to contain an additional 70Mt of tailings.

The environmental management plan for the Choco mine follows the program presented in the September 2003 environmental impact assessment report prepared by Ambioconsult, environmental consultants. The existing mine complies with all government reporting requirements. These requirements will continue with the expanded project. Compliance requires biweekly monitoring of the tailings facility, quarterly sampling of surface, subsurface and effluent waters, annual emissions monitoring, and air quality monitoring twice each year. Reporting consists of weekly and monthly internal reporting, annual

reporting to the government on waste management, monthly reporting to the Ministry for Basic Industry and Mines (MIBAM), and quarterly reporting to the Ministry of Environment (Minamb). In addition, PMG speaks to community members and workers about environmental protection as part of the company's environmental education program.

This environmental management or supervision program for the Choco mine has been expanded to include the environmental management plan requirements for the Increible 6 mines. Requirements for the environmental management program incorporate conditions of the exploitation authorization and include compliance with all applicable legislation, avoiding protected areas, waste disposal, erosion control, hazardous materials storage, fire prevention, wildlife conservation, health, safety and security, effluents, water management, community consultation, among other items.

20.4 SOCIAL AND COMMUNITY ASPECTS

20.4.1 Social Conditions

The nearest community to the mine is the village of Choco while El Callao is the closest commercial centre. There are few indigenous people in the communities surrounding the Choco 4, Choco 10, and Increible 6 concessions, and no fully indigenous communities. Indigenous communities are generally located farther south on the Rio Cuyuní. The majority of the population surrounding the operations are criolla communities, which are relatively recent, transient, and tied to small mining activities.

Cultural heritage resources have been already impacted in the area, but should be considered for any expansion activities in the future.

Relationships and consultation with the communities, workers, and small miners are important to keep the mine operating. Rusoro, as PMG, has proven its ability to maintain working relations with these groups since its acquisition of the Choco mine and the Isidora mine.

20.4.2 Social Development Initiatives

The Choco mine management has engaged in development activities in the communities near the mine and communities further away where workers live. PMG agreed to work with Corporación Venezolana de Guayana (CVG), in an addendum to the Choco 4 and Choco 10 lease agreements, to include a formal obligation to contribute at least \$250,000 per year in social contributions for the development of the mining area after the commencement of commercial production. Social programs include:

- Building of a small rural medical clinic located at the Choco Village. The clinic was constructed to specifications from the Institute of Public Health of the State of Bolivar.

- Construction of an aqueduct to supply the Choco Village with potable water.
- Resurfacing of a portion of the El Callao to Manteco road, from the EDELCA substation to the mine entrance.
- Repair of the Choco Village school.
- Local vaccination programs.
- Apprenticeship and training programs.
- Establishing local community cooperatives for the completion of company programs, including rehabilitation, construction and supply of exploration consumables.

In addition to the above, compensation agreements were made with the small miners who previously occupied the property.

The Incredible 6 exploitation permit requires 4.75M Bolivars Fuerte to be paid annually to a socio-economic fund. 70% of this fund will be used for compliance with the special provisions of the exploitation permit, and the remaining 30% of the fund go to community programs for health, education, recreation, and environmental clean-up.

20.4.3 Health and Safety

The Choco mine operates in accordance with the standard procedures, codes of safety and regulations of health and effective labour safety in Venezuela, such as Covenin Norms, LOPCYMAT and regulation of the conditions of hygiene and industrial safety.

Under the Labour Law, an employer is liable to employees or their relatives, as the case may be, for work-related accidents and occupational illnesses suffered by them, unless such accidents: (i) occur due to force majeure events; (ii) derive from the intentional will of the employee; (iii) occur while an individual is performing an occasional service for the employer which is not related to the company's business; or (iv) occur in the course of work undertaken by the employee working from his own domicile. The Labour Law provides for indemnification payments of up to two years' salary, but not in excess of 25 times the applicable monthly minimum salary. The payment of indemnification is triggered when the accident derives from the service or is directly related to it, whether or not there is fault or negligence of the employer or the employee. Furthermore, as an employer engaged in mining activities, the mine faces potential liability arising from injuries to, or deaths of, workers, including workers employed by its contractors.

The Venezuelan Organic Law of Work Conditions (the Organic Law) imposes on employers the obligation to maintain a work environment where employees are safeguarded against work-related accidents and illnesses. The Organic Law imposes certain obligations on employers which can be onerous, such as the implementation and maintenance of medical

services and the creation of employer-employee committees in charge of coordinating policies related to work and safety procedures, conditions and precautions. In case of infringement, the Organic Law provides for penalties, including criminal liability where an employee's injury results from the negligence of his or her employer or the employer's non-compliance with legal requirements.

20.5 MINE CLOSURE REQUIREMENTS

Venezuelan law requires mining operations to rehabilitate land disturbed by mining. As part of its permit obligations to the Ministry of Environment, PMG developed a reclamation and rehabilitation plan for Choco 4 and Choco 10 in 2007 titled "Propuesta Plan De Recuperación Forestal Como Medida Compensatoria A La Afectación De Recursos Por Exploración, Construcción De Facilidades Y Explotación Del Proyecto Minero Choco 4 Y Choco 10". This document outlines the objectives and specific measures to be employed for rehabilitation of each area of the mine with the focus on stabilizing slopes and revegetation with native plant species. The minimum commitment is for 12% of disturbed areas to be restored. Waste rock storage areas are to be contoured to fit the local topography and be no higher than 50 m. Slopes are to be rehabilitated in a number of ways depending on the slope, area and quality of materials. Geotextiles, hydroseeding and fertilizing will be needed on steeper slopes with poor material quality. Less than 5% legume species may be used in order to allow for natural native species to become established. A 20 cm thick organic soil layer is prescribed for some of the gentle slopes by the water reservoir.

PMG has entered into an agreement with a local university (Universidad Experimental de Guayana, UNEG) to develop a detailed decommissioning and mine closure program, including a plan for progressive rehabilitation. The Choco Mine has demonstrated success in the progressive reclamation plan, in particular with on-going reclamation of the waste rock dumps.

The mine closure and rehabilitation costs for Choco prior to the proposed expansion were estimated at \$4.8 million. The revised estimate for closure cost used in this assessment provides for an amount of \$30.1 million over the life of the operation and \$14.35 million at the end of the life of the operation.

A closure and reclamation plan is required for exploitation of the Increible 6 property. The reclamation and closure fund required by the government for Increible 6 was 14,286,749 Bolivars Fuerte.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

Expansion project capital expenditure is summarised in Table 21.1. The estimate of process and infrastructural capital was prepared by Ausenco in the third and fourth quarter, 2010. The estimate for the tailings storage facility (TSF) is included in the infrastructural capital shown below, and was prepared by Knight Piésold in mid-2010. No escalation has been applied to the estimate. The provisions for contingency in the Ausenco and Knight Piésold estimates are combined.

As of the effective date of this report, the capital estimate is believed to be accurate to $\pm 15\%$.

Table 21.1
Project Capital Expenditure Summary

Item	US\$ 000
Mining (included elsewhere)	-
Process Plant	152,831
Plant Services	14,956
Infrastructure	41,914
Design growth	14,773
Indirect costs	46,186
Contingency	40,105
TOTAL	310,765

21.1.1 Mining Capital Cost

The study has identified the fleet of mining equipment that will be required in order to achieve the scheduled production of ore and waste from the open pits. The purchase price of the fleet has also been estimated using OEM budgetary prices. However, the economic evaluation of the project reflects Rusoro's operating strategy, wherein the mining contractor is responsible for the procurement of the required fleet. In order to ensure that the cost of ownership of the expanded mining fleet is fully reflected in the cash flow model, Micon has added the appropriate finance charges to the basic contract mining rate. These leasing costs are included in the operating cash flow and do not appear in the capital cost estimate.

21.1.2 Plant Expansion Capital Expenditure

Capital expenditure for the process plant expansion is estimated as shown in Table 21.2, plant services in Table 21.3, infrastructure in Table 21.4, and indirect costs and contingency in Table 21.5.

The estimates presented in this section have been compiled by Micon from information provided by Ausenco and Knight Piésold during 2010. In Micon's opinion, the impact on these estimates of inflation and exchange rate variances between the effective date and the signature date of this report lie within the expected range of accuracy of the estimate.

Table 21.2
Plant Expansion Capital Expenditure

Item	US\$ 000
Primary Crushing and Coarse Ore Reclaim	25,212
Grinding	55,745
Preleach Thickener	6,061
Leaching Circuit	35,055
Desorption / Regeneration	9,626
Gold Room / Refinery	7,090
Tailings Handling	6,335
Reagents & Consumables General	7,709
TOTAL	152,831

Table 21.3
Plant Services Capital Expenditure

Item	US\$ 000
Air Delivery Systems	1,612
Water Delivery Systems	6,229
Utility Gallery - Pipe Rack And Cable Tray	5,566
Process Control System	1,138
Plant Area Lighting	411
TOTAL	14,956

Table 21.4
Infrastructural Capital Expenditure

Item	US\$ 000
Site Civil Infrastructure	3,250
Sewage And Waste Water Systems	517
Plant Workshop And Warehouse	1,516
Mobile Equipment (Non Mining Fleet)	1,300
Ancillary Buildings	1,670
Power Supply	5,949
HV Powerline & El Callao Substation	2,500
Tailings Transport	3,503
Tailings Storage Facility	21,709
TOTAL	41,914

Table 21.5
Indirect and Other Capital Expenditure

Item	US\$ 000
Design Growth (aggregate)	14,773
Temporary Facilities	266
Temporary Services	436
Construction Camp (50 Man Camp)	1,495
Craneage	3,350
Ocean And Local Freight And Logistics	6,890

Item	US\$ 000
EPCM Costs	30,250
Vendor Erection Supervision	814
Vendor Commissioning	488
First Fills / Spares	2,197
Contingency	40,105
TOTAL	86,291

21.1.3 Sustaining Capital

Sustaining capital expenditure for plant equipment is provided for in the cash flow model as an annual allowance of 2% of the initial cost of the expanded plant, or \$3.0 million per year. Over the LOM period these total \$36.7 million.

In addition, subsequent phases of tailings dam construction are allowed for, at a total cost of \$49.1 million, including a contingency of \$12.7 million.

Over the LOM period, the total sustaining capital expenditure amounts to \$85.8 million.

21.1.4 Closure/Reclamation Costs

A mine closure and rehabilitation provision of \$47.8 million is also provided for, made up of contributions of \$2M/y assumed over fifteen years, and a further \$17.8 million upon mine closure.

21.2 OPERATING COSTS

The cash operating costs for the expanded operation have been estimated using a zero-based approach for mining and the operation of the new process plant. Actual (past) operating costs for the Choco 10 mine have been used as a guide to future costs for the existing plant and for General and Administrative costs.

As of the effective date of this report, the operating cost estimates are believed to be accurate to $\pm 15\%$.

21.2.1 Mine Operating Costs

Mine operating costs are forecast using a zero-based estimate ($\pm 15\%$ accuracy) considering the fleet requirements in each period. Over the LOM, these rates average \$1.69/t mined for drill, blast, load and haul operations. Further allowances are made for stockpile rehandle (\$0.01/t mined) and mining supervision and technical services (\$0.07/t mined).

In addition to the above, the contractors cost of ownership of the required equipment fleet has been assumed to be passed back to the project. In order to estimate these costs, the leasing cost of the equipment has been calculated, assuming a five year leasing period on each item, using a real interest rate of 7%/y. Since the entire cost would in reality be reflected in the

contract rate, no deposits or residual amounts are considered in the cash flow model. Over the LOM, cost of ownership is estimated to add \$0.62/t mined.

21.2.2 Processing Operating Costs

Process operating costs are estimated to average \$8.34/t milled, made up as shown in Table 21.6. These estimates are derived from unit costs for labour, consumables and spare parts, and specific rates of consumption, based on throughput matched to the production schedule.

Power costs are based on a unit cost of \$0.03/kWh and an average consumption rate of 40.47 kWh/t.

Cyanide consumption is estimated at 0.54 kg/t, at a unit cost of \$3,660/t NaCN.

Table 21.6
Process Operating Costs

Item	\$/t milled
Labour	0.74
Power	0.84
Gyr/Cr Mantle (Upr + Lwr)	0.02
Gyr/Cr. (Conc & Spider)	0.03
Peb/Cr. (Mantle and Bowl)	0.01
SAG Mill Liners	0.32
Ball Mill Liners	0.04
Grinding Media	0.82
Cyanide	1.37
Lime	0.65
Flocculant	0.06
Hydrochloric Acid	0.00
Sodium Hydroxide	0.02
Anti-Scalant	0.06
Activated Carbon	0.05
Flux	0.00
Maintenance	0.91
Other (existing plant)	2.39
TOTAL	8.34

21.2.3 General and Administrative Costs

General and Administrative costs are based on experience at the operating mine, as of the effective date of this report. The average over the life of mine equates to \$3.04/t milled.

22.0 ECONOMIC ANALYSIS

22.1 BASIS OF EVALUATION

Micon has prepared its assessment of the Project on the basis of a discounted cash flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to establish the economic viability of the proposed expansion of open pit mining and processing for the production of gold from the Choco 10 and Increible 6 deposits. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

22.2 MACRO-ECONOMIC ASSUMPTIONS

22.2.1 Exchange Rate and Inflation

The project cash flow model, and all results derived from the model, are expressed in United States dollars (\$). Inputs to the cash flow model originating in Venezuelan Bolivar Fuerte (BsF) have been converted at the ruling rate². Constant, Q4-2010 money terms are used throughout, i.e., without provision for inflation.

22.2.2 Expected Metal Prices

The base case has been evaluated using trailing average metal prices to December 31, 2010, as shown in Table 22.1. In the light of upward momentum in the gold price, Micon has taken the 12-month average as its forecast price for the base case. As part of its sensitivity analysis, Micon also tested a range of prices 30% above and below the base case value, based on the 12-month trailing average.

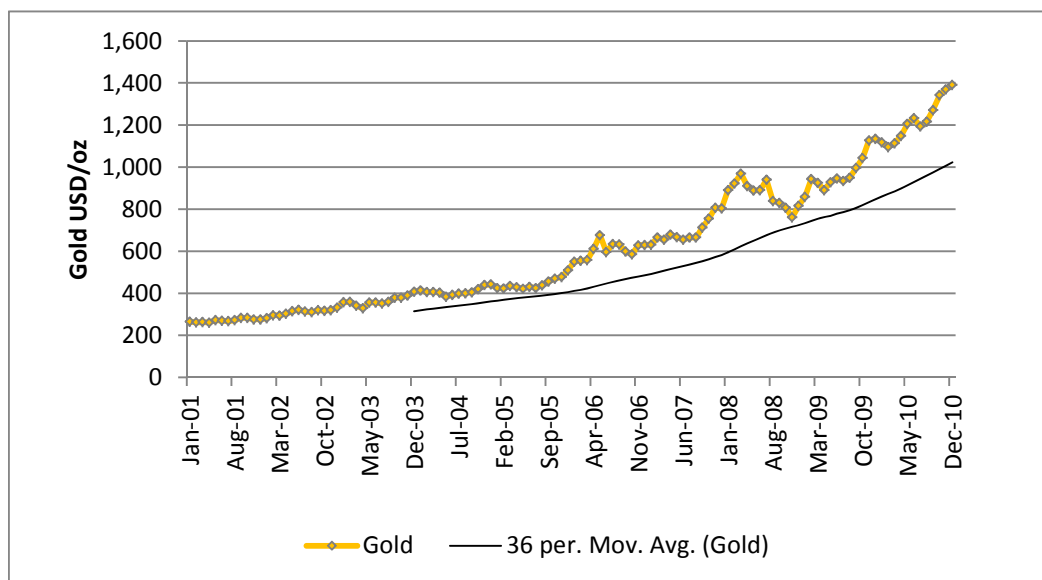
Table 22.1
Gold Price Forecasts

Item	Gold (US\$/oz)
36-month trailing average to Dec-2010	1,023
24-month trailing average to Dec-2010	1,100
12-month trailing average to Dec-2010	1,225
1-month trailing average to Dec-2010	1,391

Figure 22.1 shows the monthly average price of gold over the period 2001 to 2010 inclusive.

² As of December, 2010 Venezuela's Central Bank applies a selling rate of BsF 4.30 per US\$. Prior to January 11, 2010, the selling rate was BsF 2.15 per US\$.

Figure 22.1
Gold Price 2001-2010

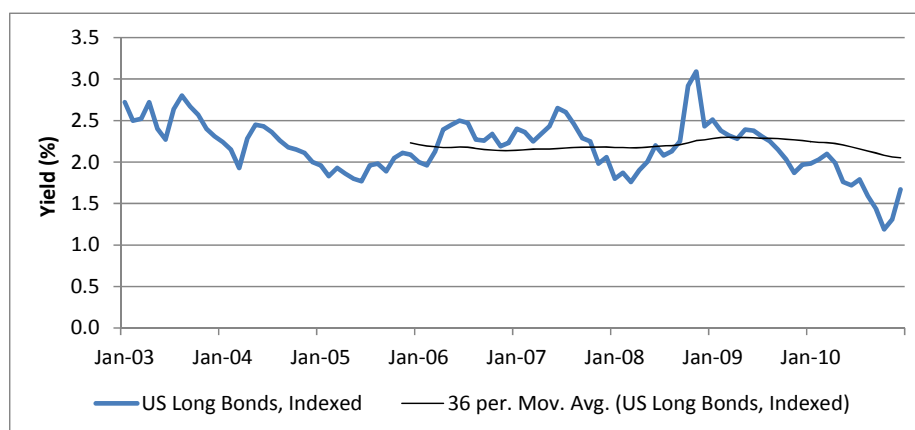


22.2.3 Weighted Average Cost of Capital

In order to find the NPV of the cash flows forecast for the project, an appropriate discount factor must be applied which represents the weighted average cost of capital (WACC) imposed on the project by the capital markets. The cash flow projections used for the valuation have been prepared on an all-equity basis. This being the case, WACC is equal to the market cost of equity.

Figure 22.2 illustrates the real return on US bonds computed by the Federal Reserve Board, taken as a proxy for the risk-free interest rate.

Figure 22.2
Real Return on US Long Bonds 2003-2010
(Source: US Federal Reserve)



It will be noted that over much of the past seven years this rate has fluctuated between 1.5% and 2.5% and, although it has recently fallen below 1.5%, the average is close to 2.0% over this period.

Historically, the risk premium for equity has been estimated at 5.0%. The value of beta (β) for similar gold producers is observed to lie in the range 0.6 to 1.4 so, applying the Capital Asset Pricing Model (CAPM), the project's cost of equity will lie in the range of 5.0% to 9.0%. Accordingly, Micon has selected a discount rate of 7.0% as its base case estimate for the project's cost of equity and the economic results are presented for discount rates ranging from 5.0% to 9.0%.

22.2.4 Taxation Regime

Venezuelan corporate income tax has been allowed for at the rate of 34%. Expansion project capital expenditure for the establishment of the new processing capacity, taken together with ongoing or sustaining capital, is assumed to be eligible for depreciation based on the unit of production method over the remaining life of the mine. The effective tax rate is thus approximately 28%.

22.2.5 Royalty

A royalty of 3.0% has been provided for in the cash flow model.

22.2.6 Marketing Costs

Costs relating to the refining and disposal of gold production take into account the market structure in Venezuela, in which 50% of the total sales are earmarked for sale to the Central Bank of Venezuela at the ruling exchange rate, and the remainder for export. For the purposes of this study, it is assumed that all cash flows are convertible to US dollars at the ruling exchange rate.

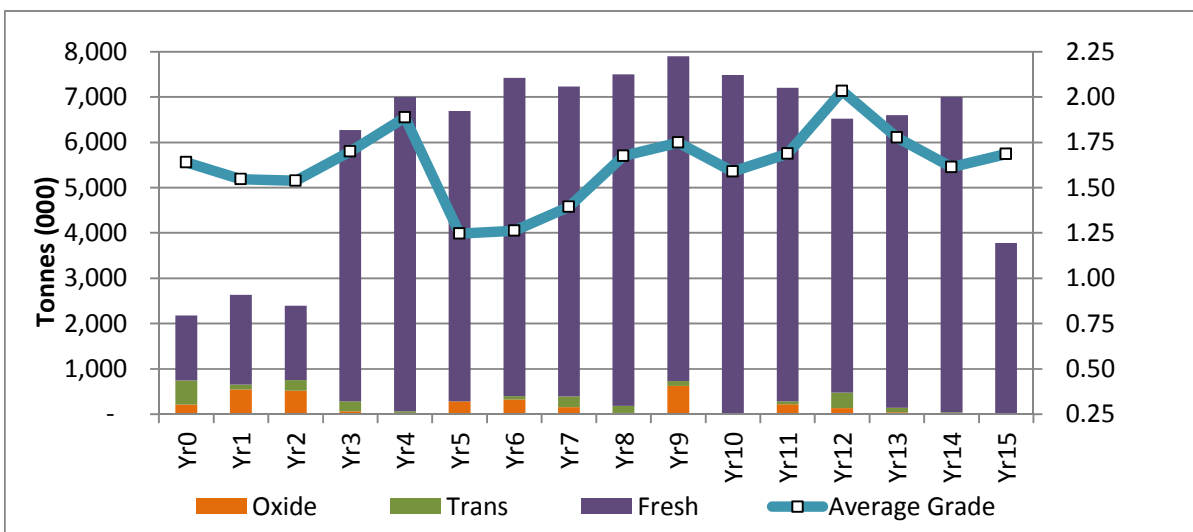
22.3 TECHNICAL ASSUMPTIONS

The technical parameters, production forecasts and estimates described earlier in this report are reflected in the base case cash flow model. These inputs to the model are summarised below.

22.3.1 Mine Production Schedule

In Years 0 to 2, the current mill is fed at the rate of 5,000 t/d, comprising a mixture of oxide, transition and fresh ore types. During this period, waste mining ramps up to expose additional ore in anticipation of the expansion of milling capacity. After Year 3, the rate of ore mining increases, ROM tonnage being dominated by fresh ore, with remnant oxides being mined during stripping of the new open pits, including the Incredible area. Figure 22.3 shows the annual ore mining schedule.

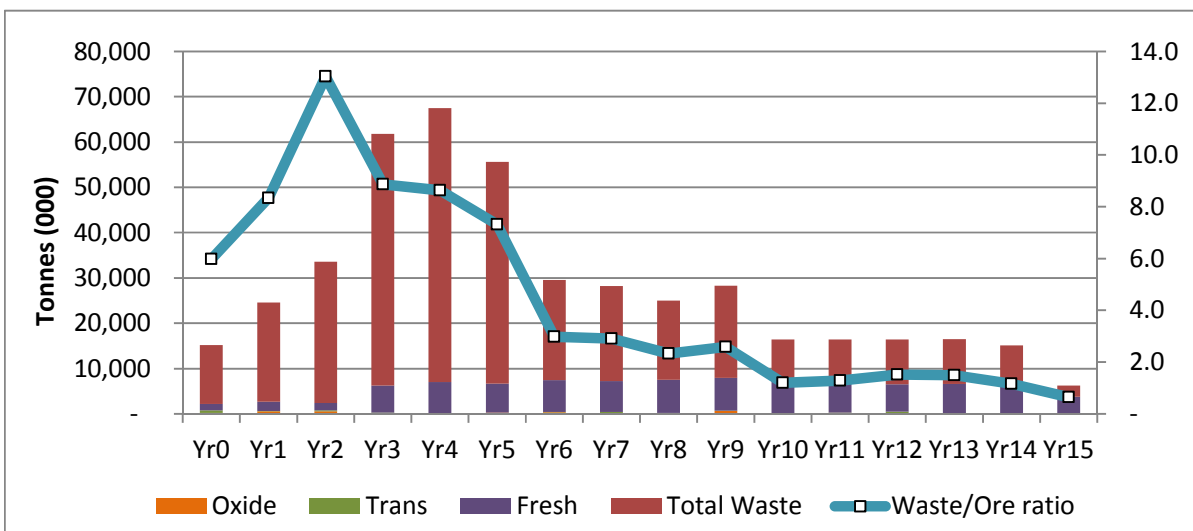
Figure 22.3
Ore Mining Production Schedule



The grade profile reflects the result of detailed optimization of the mining schedule, whereby waste stripping is carried out as late as possible, consistent with maximising mill feed grade, and keeping steady the required mining fleet capacity while maintaining ore availability and fleet utilization.

Figure 22.4 shows the combined ore and waste mining schedule. Waste mining ramps up during construction of the new milling capacity, peaking in Year 4 when the VBK pit is being opened up, before falling back sharply in Year 6 and thereafter.

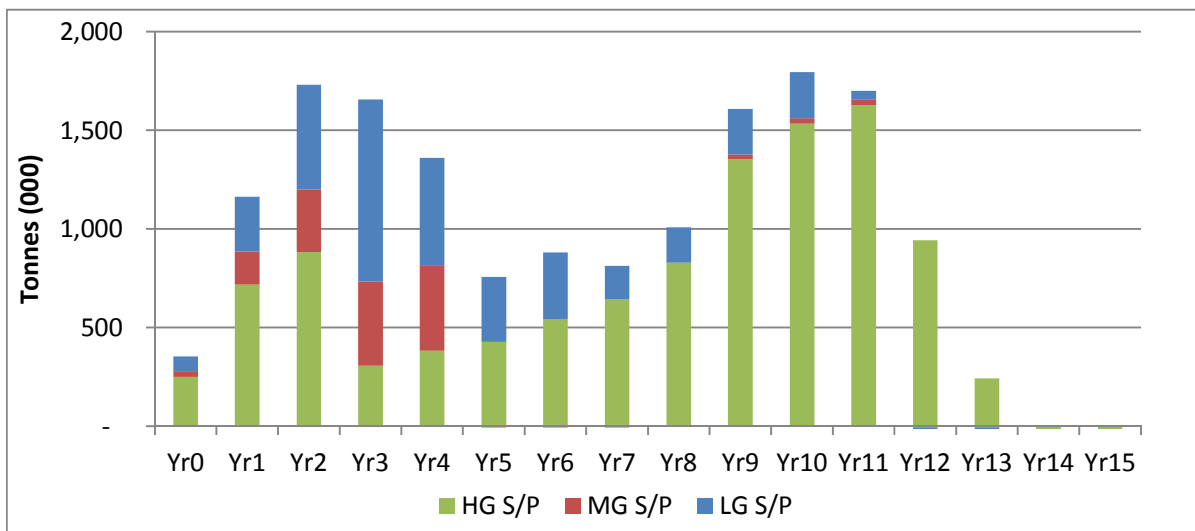
Figure 22.4
Ore and Waste Mining Production Schedule



22.3.2 Stockpiling

Three stockpiles are accounted for in the production schedule, designated by grade band (high, medium and low). The closing balances of each grade band are shown in Figure 22.5. It will be seen that once the expansion project is complete, the stockpile fluctuates between 800,000 and 1,800,000 tonnes, equivalent to between 40 and 90 days of plant feed.

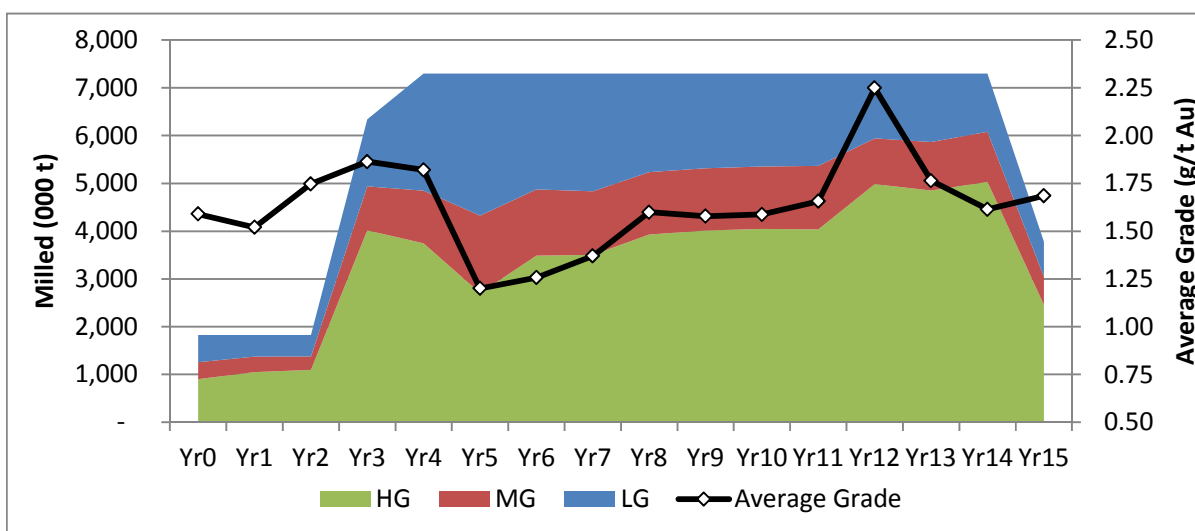
Figure 22.5
Ore Stockpiles – Closing Balances



22.3.3 Processing Schedule

Annual tonnage and grade of ore milled is shown in Figure 22.6.

Figure 22.6
LOM Processing Schedule and Grade Profile

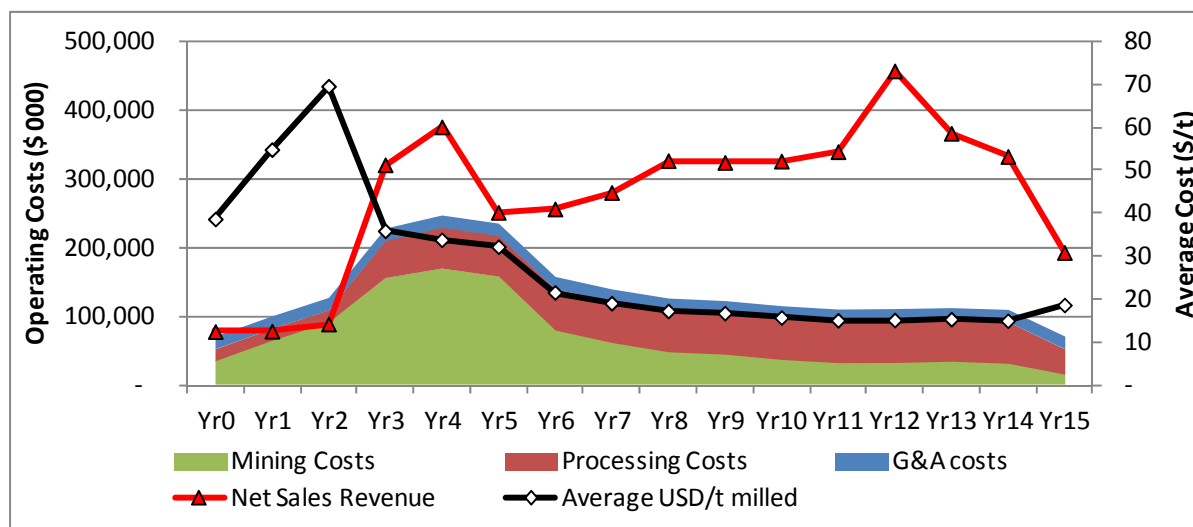


Following expansion of the process plant, in Year 3 significant quantities of low grade material are milled while the VBK open pit is being opened up. As the VBK pit develops, the proportion of high grade ore in the mill feed slowly increases, raising the average grade of ore milled from 1.25 g/t to around 1.75 g/t and then 2.25 g/t in Year 12.

22.3.4 Operating Costs

Cash operating costs average \$22.73/t milled over the LOM period, including \$11.36/t mining, \$8.34/t processing and \$3.04/t general and administrative costs. The mining cost estimate includes a provision for increases in the contractors' unit rate to cover leasing charges in respect of the expanded mining fleet and its periodic replacement. This operating expense is in lieu of capital expenditure of approximately \$153 million for new mining equipment over the LOM period. Figure 22.7 shows the operating expenditures over the LOM period. The average cost per tonne milled follows closely the average stripping ratio seen in Figure 22.4 (above). The initially high waste:ore ratios put pressure on operating margins in the first few years, but after Year 5, operating margins are increasingly robust over the remainder of the LOM.

Figure 22.7
Cash Operating Costs

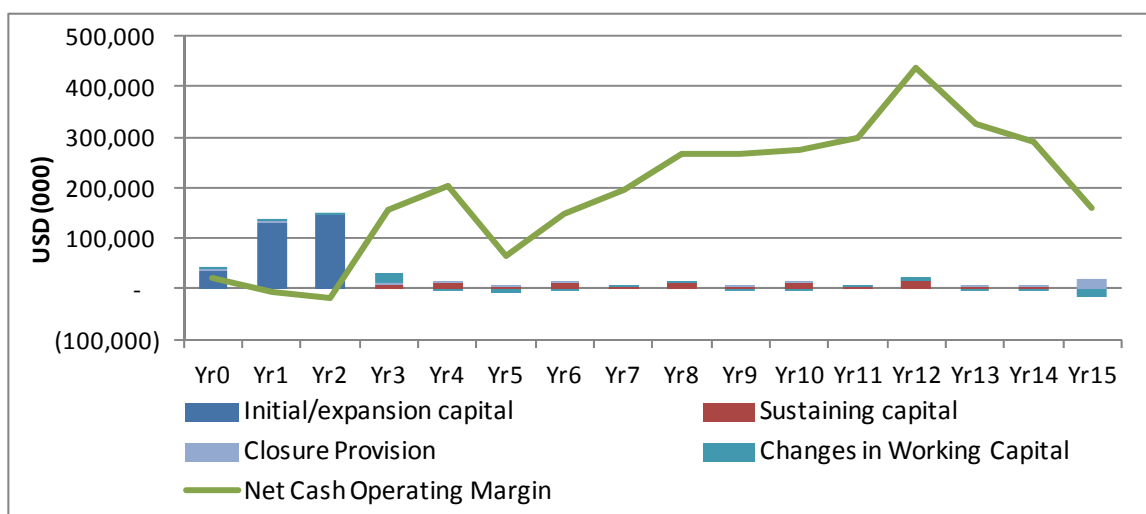


22.3.5 Capital Costs

Capital expenditures for the expansion project are estimated to total \$310.8M, including \$152.8M processing, \$15.0M for services, \$41.9M infrastructure, \$14.8M for design growth, \$46.2M indirect costs and a contingency of \$40.1M, equivalent to almost 18% of the direct capital cost estimate. In addition, sustaining capital has been provided for in the amount of \$85.8M over the LOM period. A mine closure and rehabilitation provision of \$47.8 million is also provided for, made up of contributions of \$2M/y assumed over fifteen years, and a further \$17.8 million upon mine closure.

Working capital has been estimated to include 15 days product inventory comprising the mill, leaching circuit, carbon and elution inventories, and 15 days accounts receivable. Stores provision is for 60 days of consumables and spares inventory, less 30 days accounts payable. A balance of \$14.25M in working capital is assumed to be brought forward from the current operation. Figure 22.8 compares annual capital expenditures over the preproduction and LOM periods with the project's cash operating margin.

Figure 22.8
Capital Expenditures



22.3.6 Project Cash Flow

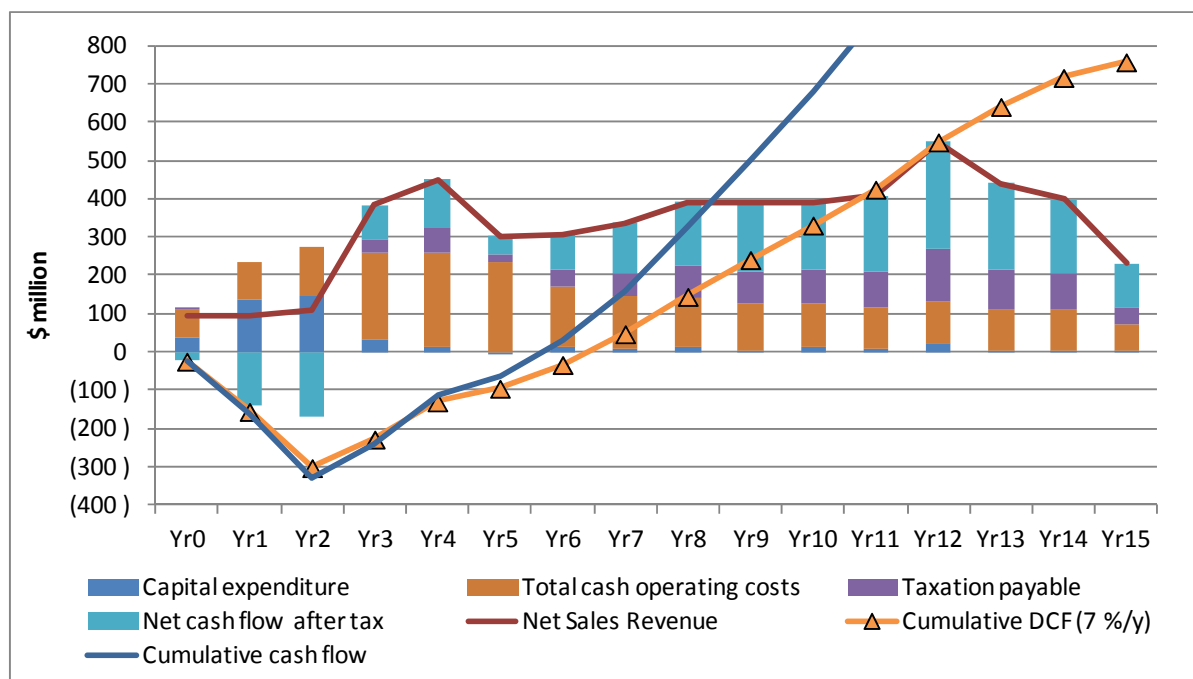
The LOM base case project cash flow is summarised in Table 22.2

Table 22.2
Life-of-Mine Cash Flow Summary

	LOM Total (\$M)	\$/t Milled	\$/oz Gold
Gross Revenue	5,481.6	57.16	1,225.00
Selling Exp. & Royalty	(218.4)	(2.28)	(48.81)
Net revenue	5,263.2	54.88	1176.19
Mining costs	1,089.3	11.36	243.44
Processing costs	799.5	8.34	178.68
General & administrative costs	291.4	3.04	65.13
Total cash operating cost	2,180.3	22.73	487.24
Net operating margin	3,082.9	32.15	688.94
Capital expenditure	444.3	4.63	99.30
Net cash flow (before tax)	2,638.5	27.51	589.64
Taxation	952.4	9.93	212.85
Net cash flow (after tax)	1,686.1	17.58	376.80

Annual cash flows are presented in Figure 22.9 and Table 22.3 (over).

Figure 22.9
Life-of-Mine Cash Flows



22.4 BASE CASE EVALUATION

In addition to the expansion capital of \$310.8M, the project has additional funding requirements for the acceleration of waste mining and increases in working capital, bringing the cumulative cash flow at the end of Year 2 to (negative) \$331.2M. From that point, with completion of the expansion project, the project demonstrates an undiscounted pay back of 3.7 years or approximately 4.4 years when the cash flow is discounted at the selected rate of 7.0%. The latter leaves a production tail of 8.0 years on the current mineral reserve.

Over the LOM period, the average cash operating cost equates to \$487/oz gold.

The base case evaluates to an IRR of 40% before tax and 30% after tax. At the selected discount rate of 7.0%, the net present value (NPV₇) of the cash flow is \$1260.7M before tax and \$758.0M after tax.

The base case cash flow evaluation results are shown in Table 22.4, in which results at the comparative discount rates of 5%/y and 9%/y are also presented.

Table 22.3
Base Case Life of Mine Annual Cash Flow

		TOTAL	Yr0	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15
Ore Production																		
Oxide	000t	3,247	211	547	521	63	29	278.9	322.3	155.5	26.2	687.0	1.0	217.1	135.3	44.1	7.5	-
Trans	000t	2,228	532	105	229	217	30	-	75.8	239.9	152.6	104.9	-	63.4	342.5	99.0	35.5	-
Fresh	000t	90,411	1,435	1,983	1,642	5,989	6,943	6,412.8	7,024.9	6,836.9	7,319.8	7,174.5	7,485.4	6,925.5	6,045.9	6,457.9	6,960.6	3,775.8
Total Ore Mined	000t	95,886	2,178.5	2,634.8	2,392.9	6,268.3	7,002.5	6,691.7	7,423.0	7,232.4	7,498.7	7,966.4	7,486.4	7,206.0	6,523.7	6,601.0	7,003.6	3,775.8
Ore Grades																		
Oxide	g/t Au	1.48	1.20	1.18	1.22	0.83	1.41	1.15	1.83	2.75	1.83	1.65	1.22	1.48	1.90	1.38	1.86	-
Trans	g/t Au	1.67	1.92	1.35	1.13	0.98	2.18	-	1.07	1.88	2.80	1.56	-	1.18	1.64	2.06	1.71	-
Fresh	g/t Au	1.63	1.60	1.66	1.70	1.74	1.89	1.25	1.24	1.34	1.65	1.76	1.59	1.70	2.06	1.78	1.61	1.68
Average Grade	g/t Au	1.63	1.64	1.55	1.54	1.70	1.89	1.25	1.26	1.39	1.67	1.75	1.59	1.69	2.03	1.78	1.61	1.68
Total Waste	000t	361,034	13,018	21,959	31,192	55,556	60,490	48,955	22,142	21,019	17,467	20,760	8,939	9,219	9,901	9,869	8,107	2,441
Waste/Ore ratio	t/t	3.8	6.0	8.3	13.0	8.9	8.6	7.3	3.0	2.9	2.3	2.6	1.2	1.3	1.5	1.5	1.2	0.6
Ore treated	000 tonnes	95,902	1,825	1,825	1,825	6,342	7,300	7,300	7,300	7,300	7,300	7,301	7,300	7,300	7,300	7,300	7,300	3,785
	Au grade (g/t)	1.63	1.59	1.52	1.75	1.86	1.82	1.20	1.26	1.37	1.60	1.58	1.59	1.66	2.25	1.76	1.61	1.68
	Au content (000 oz)	5,022.2	93.3	89.2	102.5	380.0	427.2	281.5	295.0	321.4	375.3	370.0	372.5	388.8	527.9	413.9	378.7	205.0
	Recovery (%)	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0
	Au yield (000 oz)	4,520.0	83.9	80.2	92.3	342.0	384.5	253.4	265.5	289.3	337.7	333.0	335.2	349.9	475.1	372.5	340.9	184.5
Gold inventory	Au (000 oz)		3.6	3.0	3.5	16.0	14.2	9.1	10.9	11.9	13.9	13.7	13.8	14.4	19.5	15.3	14.0	-
Gold sales (gross)	Au (000 oz)	4,520.0	80.3	80.9	91.7	329.6	386.3	258.5	263.7	288.3	335.7	333.2	335.2	349.3	470.0	376.7	342.2	198.5
Gold payability	%	99%	99%	99%	99%	99%	99%	99%	99%	99%	99%	99%	99%	99%	99%	99%	99%	99%
Gold Sales (payable)	Au (000 oz)	4,474.8	79.5	80.1	90.8	326.3	382.4	255.9	261.0	285.4	332.4	329.9	331.8	345.8	465.3	372.9	338.7	196.5
Revenue (US\$ 000)	Gross Sales	5,481,588	97,399	98,125	111,204	399,718	468,454	313,500	319,783	349,631	407,170	404,081	406,456	423,622	569,935	456,829	414,944	240,737
	less Bullion transport	(2,487)	(162)	(162)	(162)	(162)	(162)	(159)	(159)	(159)	(159)	(159)	(159)	(159)	(159)	(159)	(159)	(84)
	less Refining charges	(13,424)	(239)	(240)	(272)	(979)	(1,147)	(768)	(783)	(856)	(997)	(990)	(995)	(1,037)	(1,396)	(1,119)	(1,016)	(590)
	less Insurance	(5,482)	(97)	(98)	(111)	(400)	(468)	(313)	(320)	(350)	(407)	(404)	(406)	(424)	(570)	(457)	(415)	(241)
	less Marketing	(34,260)	(609)	(613)	(695)	(2,498)	(2,928)	(1,959)	(1,999)	(2,185)	(2,545)	(2,526)	(2,540)	(2,648)	(3,562)	(2,855)	(2,593)	(1,505)
	Net Invoice Value	5,425,935	96,292	97,011	109,964	395,679	463,749	310,297	316,522	346,081	403,062	400,003	402,355	419,354	564,248	452,239	410,761	238,318
	Royalty	(162,778)	(2,889)	(2,910)	(3,299)	(11,870)	(13,912)	(9,309)	(9,496)	(10,382)	(12,092)	(12,000)	(12,071)	(12,581)	(16,927)	(13,567)	(12,323)	(7,150)
	Net Sales Revenue	5,263,157	93,403	94,101	106,665	383,808	449,836	300,988	307,026	335,699	390,970	388,003	390,284	406,773	547,321	438,672	398,438	231,168
Cash op. costs (US\$ 000)	Mining Costs	1,089,342	35,112	64,551	91,051	155,262	168,924	157,352	79,535	61,528	48,001	44,660	36,932	32,326	32,612	34,379	31,490	15,629
	Processing Costs	799,537	17,230	17,230	17,640	54,228	59,785	59,545	59,785	59,573	59,785	59,548	59,785	59,573	59,785	59,545	59,785	36,718
	G&A costs	291,423	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214	18,214
	Total cash operating costs	2,180,303	70,556	99,995	126,905	227,704	246,922	235,111	157,533	139,315	125,999	122,423	114,930	110,113	110,610	112,138	109,488	70,561
Net Cash Operating Margin	(US\$ 000)	3,082,854	22,847	(5,894)	(20,240)	156,105	202,914	65,877	149,493	196,384	264,971	265,581	275,354	296,661	436,711	326,535	288,949	160,608
Capital Expenditure	Initial/expansion capital	310,765	35,294	130,815	144,655	-	-	-	-	-	-	-	-	-	-	-	-	-
	Sustaining capital	85,806	-	-	-	8,524	11,851	3,057	10,896	3,057	10,972	3,057	10,563	3,057	14,661	3,057	3,057	-
	Closure Provision	47,772	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	17,772
	Changes in Working Capital	0.000000	1,103	1,639	1,274	19,800	(1,166)	(7,194)	(1,792)	280	1,615	(298)	(276)	440	5,800	(4,390)	(1,784)	(15,052)
Net cash flow before tax	(US\$ 000)	2,638,511	(15,550)	(140,348)	(168,169)	125,781	190,229	68,014	138,389	191,048	250,384	260,822	263,067	291,164	414,249	325,868	285,677	157,887
Taxation payable		952,438	7,857	-	-	34,520	62,900	18,162	45,525	61,728	82,983	84,134	85,702	93,948	136,299	102,976	90,631	45,073
Net cash flow after tax	(US\$ 000)	1,686,073	(23,407)	(140,348)	(168,169)	91,261	127,329	49,852	92,864	129,320	167,401	176,688	177,365	197,216	277,950	222,892	195,046	112,813
Cumulative cash flow			(23,407)	(163,755)	(331,924)	(240,663)	(113,334)	(63,482)	29,381	158,701	326,102	502,791	680,156	877,372	1,155,322	1,378,213	1,573,259	1,686,073
Payback period on undiscounted cash flow (years)		3.7				1.00	1.00	1.00	0.68	-	-	-	-	-	-	-	-	-
Discounted Cash Flow (7 %/y)		757,964	(23,407)	(131,166)	(146,885)	74,496	97,139	35,544	61,879	80,534	97,429	96,107	90,163	93,696	123,413	92,492	75,642	40,889
Cumulative DCF (7 %/y)			(23,407)	(154,574)	(301,459)	(226,963)	(129,824)	(94,280)	(32,401)	48,133	145,561	241,668	331,832	425,528	548,941	641,433	717,075	757,964
Payback period on discounted cash flow (years)		4.4				1.00	1.00	1.00	1.00	0.40	-	-	-	-	-	-	-	-
Capital expenditure		444,343	38,398	134,454	147,929	30,324	12,685	(2,137)	11,104	5,336	14,586	4,758	12,287	5,497	22,462	667	3,272	2,721
Ave Revenue per tonne treated		54.88	51.18	51.56	58.45	60.52	61.62	41.23	42.06	45.99	53.56	53.15	53.46	55.72	74.98	60.09	54.58	61.08
Ave Cost per tonne treated		22.73	38.66	54.79	69.54	35.90	33.82	32.21	21.58	19.08	17.26	16.77	15.74	15.08	15.15	15.36	15.00	18.64
Operating Margin		58.6%	24.5%	-6.3%	-19.0%	40.7%	45.1%	21.9%	48.7%	58.5%	67.8%	68.4%	70.6%	72.9%	79.8%	74.4%	72.5%	69.5%
US\$ Cost per ounce payable gold		487.24	887.40	1,248.34	1,397.96	697.84	645.70	918.70	603.47	488.12	379.08	371.13	346.38	318.42	237.74	300.70	323.23	359.05

Table 22.4
Base Case Cash Flow Evaluation

\$ million	LOM Total	Discounted at 5%/y	Discounted at 7%/y	Discounted at 9%/y	IRR (%)
Gross Sales	5,481.6	3,673.4	3,179.7	2,775.4	
Cost of Sales and Royalty	(218.4)	(146.5)	(126.8)	(110.7)	
Net Sales Revenue	5,263.2	3,526.9	3,052.8	2,664.6	
Mining Costs	1,089.3	841.3	767.1	703.2	
Processing Costs	799.5	542.4	471.6	413.5	
G&A costs	291.4	207.3	184.1	165.0	
Total cash operating costs	2,180.3	1591.0	1,422.8	1,281.7	
Net Operating Margin	3,082.9	1936.0	1,630.0	1,383.0	
Capital expenditure	444.3	387.2	369.3	353.6	
Net cash flow before tax	2,638.5	1548.8	1,260.7	1,029.4	40%
Taxation payable	952.4	597.4	502.7	426.3	
Net cash flow after tax	1,686.1	951.4	758.0	603.1	30%

In Micon's opinion, the results demonstrate the economic viability of the project base case, under the conditions described above.

22.5 SENSITIVITY STUDY

22.5.1 Sensitivity to Revenue Factors, Operating and Capital Costs

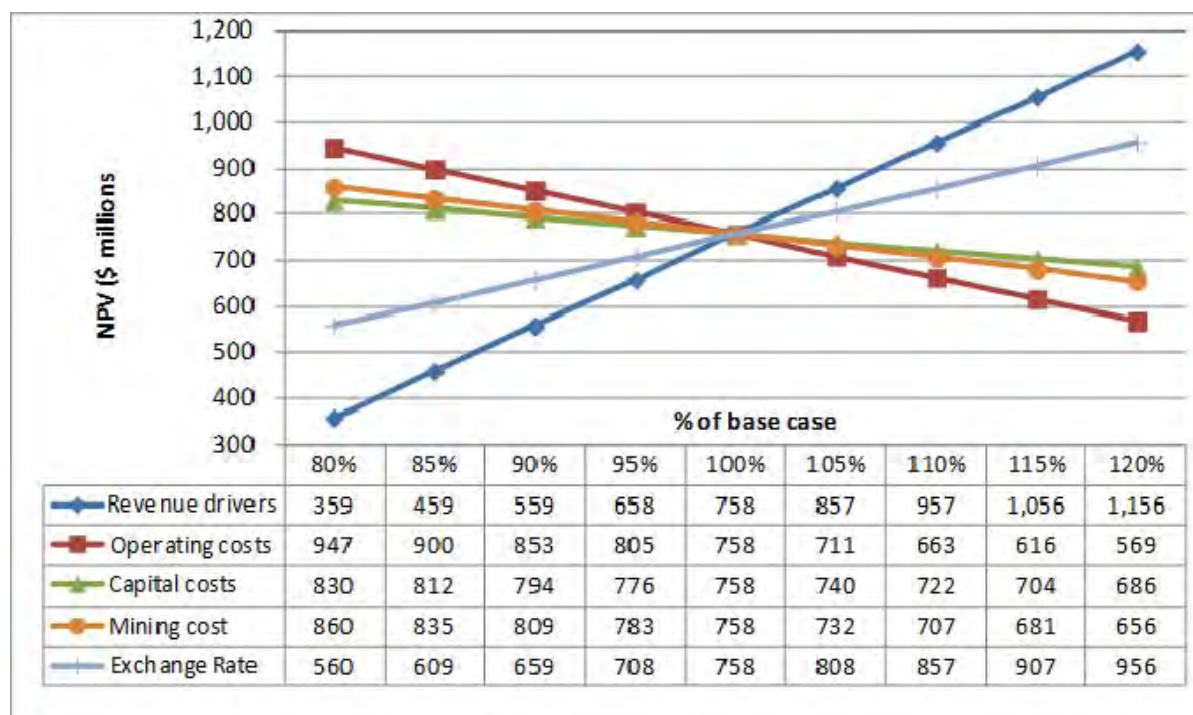
The sensitivity of the project returns to changes in all revenue factors (including grades, recoveries, prices and exchange rate assumptions) together with capital and operating costs was tested over a range of 20% above and below base case values. Figure 22.10 shows the results of this analysis.

The results show that the project is most sensitive to revenue factors, with an adverse change of 20% reducing NPV₇ by \$399M to \$359M. The project is less sensitive to operating costs, with an adverse change of 20% reducing NPV₇ by \$189M to \$569M. Capital costs have the least impact on returns, with a 20% increase in cost reducing NPV₇ by \$72M to \$686M.

In further analysis, Micon notes that simultaneously applying an increase of 57% to both capital and operating costs simultaneously would be required in order to reduce NPV₇ to zero. Likewise, a gold price of \$766/oz is required to produce a zero NPV₇, i.e., an economic break-even.

Micon concludes that the project is sufficiently robust to withstand adverse changes in the principal value drivers of the project, within the limits of accuracy of the estimate.

Figure 22.10
Sensitivity to Capital, Operating Costs and Revenue



22.5.2 Gold Price and Discount Rate

The sensitivity of project returns to changes in the gold price and discount rate were determined, as shown in Table 22.5.

Table 22.5
Sensitivity to Gold Price and Discount Rate

Gold Price		NPV (\$ million)		
\$/oz	Averages as of 31 Dec 2010	Discounted at 5.0%	Discounted at 7.0%	Discounted at 9.0%
1,000		528.9	392.1	283.5
1,023	36-month ave	572.3	429.7	316.4
1,100	24-month ave.	716.8	554.8	425.7
1,200		904.5	717.3	567.6
1,225	12-month ave.	951.4	758.0	603.1
1,300		1,092.2	879.8	709.5
1,391	1-month ave.	1,262.9	1,027.6	838.4
1,500		1,467.2	1,204.4	992.7
1,600		1,654.6	1,366.5	1,134.1
1,700		1,841.9	1,528.5	1,275.4
1,800		2,029.2	1,690.5	1,416.7
1,900		2,216.5	1,852.5	1,558.0
2,000		2,403.8	2,014.5	1,699.3

22.6 CONCLUSION

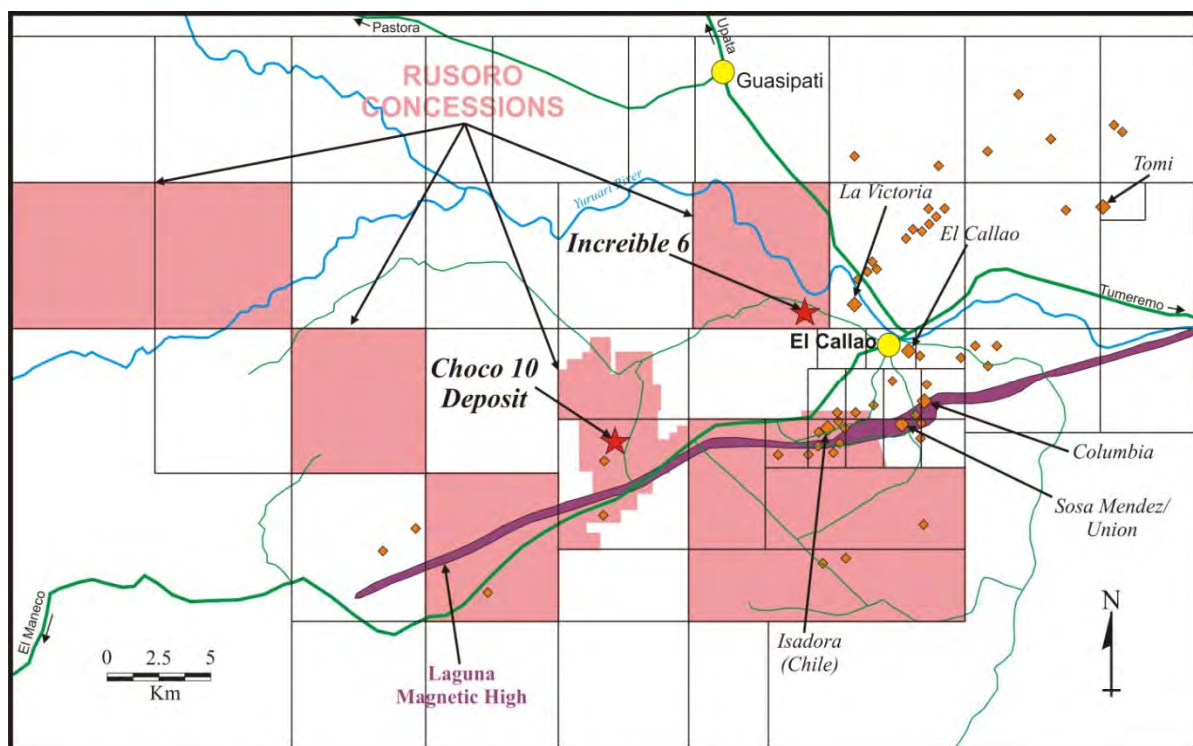
Micon concludes that this study demonstrates the viability of the project as proposed, and that expansion of the mine to process 20,000 t/d is warranted.

23.0 ADJACENT PROPERTIES

In the El Callao Gold district, the gold deposits are aligned in a broad east-northeast trend which mimics the Guri Fault trend to the north (see Figure 7.2). It is noteworthy that most major deposits lie close to and/or to the north of the Laguna magnetic high as illustrated in Figure 15.1, a feature which is believed to mark an important structure in the greenstone. Most of the deposits are hosted in the volcanic and volcanoclastic rocks of the Pastora Supergroup (e.g. Choco 10, Colombia, Sosa Mendes/Union and Isidora) and less significantly in the overlying Botanamo Group (e.g. Tomi).

Gold deposits in the district highlight the potential for both high-grade, low-tonnage deposits such as the historical El Callao mine and the Isidora mine (formerly Mina Chile) as well as low-grade, higher-tonnage deposits such as La Victoria and Choco 10.

Figure 23.1
El Callao Gold District - Adjacent Properties Map



The following information on adjacent properties is taken from publicly available documents, but Micon has been unable to verify this information. (Published resource figures vary depending on the source and what exploration was completed.) The information presented below is not necessarily indicative of the Choco 10 or Incredible 6 deposits.

23.1 LA VICTORIA AND TOMI MINES

The La Victoria and Tomi mines of Crystallex International Corporation are located approximately 25 and 40 km, respectively, northeast of the Choco 10 mine. These mines ceased operation when Crystallex's Revemin mill was handed over to Minerven, a Venezuelan state-owned company in October 1, 2008. The gold production for 2006 to 2008 is illustrated in Table 15.1.

Table 23.1
La Victoria and Tomi, Production

Category	Gold (ounces)			
	2008	2007	2006	Total
Tomi Open Pit	6,530	8,437	19,428	34,395
Tomi Underground	3,856	13,551	22,210	39,617
La Victoria	6,488	9,356	3,260	19,104
Total Production	16,874	31,344	44,898	93,116

Source : www.crystallex.com

23.2 ISIDORA MINE (MINA CHILE)

The Isidora mine is located approximately 5 km east of the Choco 10 mine. The mine is a 50/50 joint venture with the Venezuelan government and Rusoro. The mine commenced production in 2005 and has used the Choco 10 mill to process the ore for the last several years. It has produced approximately 250,000 ounces of gold. A NI 43-101 report was completed on the mineral resources as of March 31, 2008 and is illustrated in Table 15.2 (Source: www.rusoro.com).

Table 23.2
Isidora Mine Mineral Resources, March 31, 2008

Category	Tonnes	Grade (g/t Au)	Gold (ounces)
Measured	107,534	29.06	100,495
Indicated	362,738	19.75	230,376
M + I	470,272	21.88	330,871
Inferred	99,038	14.13	45,006

Source : Leader et. al., August 1, 2008

The mineral reserves reported by Hecla Mining Ltd. As of December 31, 2007 are listed in Table 15.3.

Table 23.3
Isidora Mine Mineral Reserve, March 31, 2008

Category	Tonnes	Grade (g/t Au)	Gold (ounces)
Proven	70,298	37.20	84,029
Probable	109,149	28.80	101,056
Proven and Probable	179,447	32.10	185,085

Source : Leader et. al., August 1, 2008

23.3 COLUMBIA MINE

The Columbia mine is located approximately 25 km east of Choco 10 near the town of El Callao. It is owned by CVG-Minerven, a Venezuelan state-owned company).

Production in 2004 was 100,906 ounces (256,339 t at 13 g/t Au) and reserves were stated at 1.6 Mt at a grade of 9.2 g/t Au with ore shoots grading up to 60 g/t Au, (Channer, Graffe, and Vielma, 1996 – SEG Newsletter).

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT SCHEDULE

A Level 2 schedule has been developed for the project covering all major activities from completion of the study through to process plant ramp-up. It also includes key milestones for the project, some of which are imposing constraints. The schedule consists of a logic-networked critical-path schedule based on all known project requirements and the proposed methods of project implementation.

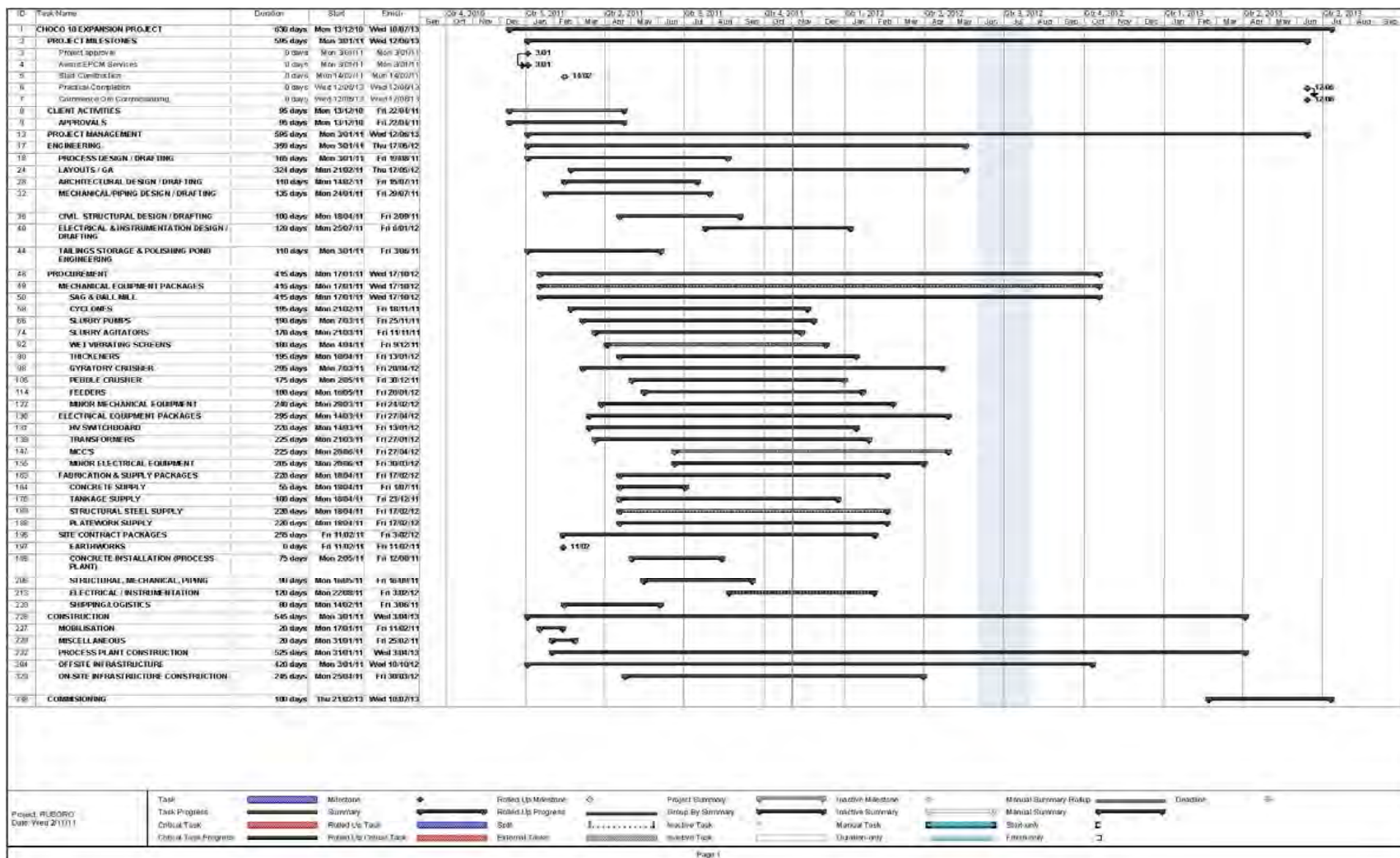
The implementation schedule is based on delivery of the SAG and ball mills 55 weeks from the date of placing the order, assumed to be 18 weeks after Board approval to proceed. The scheduled mill installation and commissioning period is 37 weeks. The mills are likely to be the critical item of equipment required to complete the project, they will therefore be tendered and a recommendation to purchase issued as soon as practical after the commencement of sufficient engineering. Other long-lead items such as transformers and the primary crusher will also need to be ordered near the commencement of engineering. The source of permanent site power for the expansion has not been definitively resolved for this study, and the schedule impact of the power supply options is uncertain. Supply of permanent power has the potential to become the schedule critical path.

24.2 PROJECT IMPLEMENTATION

Figure 24.1 shows a summary of the project implementation schedule, based on a notional start date at the end of 2010, valid as of the effective date of this report. However, this schedule will need to be re-validated once PMG's approval for project implementation is given, and Micon notes that, as of the date of signing this report, slippage of one year had already occurred in the project implementation schedule.

In particular, changes in the lead time for delivery of items such as transformers, the gyratory crusher, SAG and ball mills may have a material impact on the duration of the construction period and the eventual date of commissioning the expanded plant.

Figure 24.1
Project Implementation Schedule



25.0 INTERPRETATION AND CONCLUSION

Rusoro Mining Ltd. has successfully been able to identify additional mineral resources within the Choco 10 concession, over the last four years. It has also updated the Incredible mineral resources.

The current mineral resource estimate for all eight deposits is 162.3 million tonnes of mineralization with an average grade of 1.85 g/t gold in the measured and indicated categories. This is equivalent to 9.7 million ounces of gold in-situ. There is a total inferred mineral resource of 68.7 million tonnes with an average grade of 1.48 g/t gold. It cannot be assumed that all or any part of this Inferred Mineral Resource will be upgraded to an indicated or measured mineral resource as a result of continued exploration. Confidence in the inferred mineral resource is insufficient to allow the meaningful application of technical and economic parameters.

The majority of the increased tonnage is derived from the VBK deposit where most of the 406 new drill holes were completed. This deposit and its mineral resources is an order of magnitude larger than the other three deposits. There has been no stripping or development mining undertaken at this deposit.

The VBK deposit is a series of stacked vein systems which have good continuity along strike and down the dip. "Domain 41" is the largest of the vein systems within VBK with respect to tonnage. The grade of the VBK veins systems appear to increase at depth. Mineralization is still open at depth at VBK

Even though the Rosika, Coacia and Pisolita deposits have been mined over the last five years, additional mineralization has been identified in all three deposits. Mineralization has been identified either along strike or at depth. Due to the close proximity of all four deposits, their extremities overlap each other. Drilling one deposit, especially at depth, usually intersects another deposit, hence increasing mineral resources of both deposits. Due to close-spaced drill holes some mineralization has been transferred to higher confidence mineral resource categories. Mineralization is still open at depth at Rosika and possibly Coacia

Drilling between Coacia and Pisolita indicate that the Pisolita may be the up-dip extension of the Coacia deposit. The deeper Coacia vein systems flatten as they approach the surface and may be traced into mineralized vein systems within the flat lying Pisolita deposit.

The Incredible 6 deposits show a slight decrease in estimated tonnage and grade due to difference in interpolation and interpretations of the deposits. The excellent grade control procedures that are done within the RCP open pits will most likely increase the tonnage of the Incredible deposits.

Micon concludes that this study demonstrates the viability of the project as proposed, and that expansion of the mine to process 20,000 t/d is warranted.

26.0 RECOMMENDATIONS

26.1 GEOLOGY

It is recommended that a drill program of approximately 50 diamond drill holes be located in strategic areas, within the VBK deposit. The holes would range between 240 and 686 m. The total depth of all holes would be approximately 25,000 m. This program should take approximately two to three months, if there are at least two drill rigs working on the property. The anticipated budget would be approximately US\$ 5,000,000 at an “all in” cost rate of US\$ 200 per metre.

It is recommended that all diamond drill holes be collared HQ size (63.5 m) and if necessary reduced to NQ size (47.6 m). It is recommended that all Rusoro drilling and sampling procedures be followed to provide appropriate QA/QC controls and database validation.

The main purpose of these holes would be to continue to target areas where there is wide-spaced drilling and Inferred Mineral Resource within the proposed VBK open pit boundary. The program is designed to upgrade these areas to either an Indicated or Measured Mineral Resource category. Some of the proposed holes will be specifically designed to provide geotechnical data for future pit slope design and mine planning. The program will also provide additional sample material for future metallurgical testwork on the VBK deposit mineralization. The results of this metallurgical testwork would most likely be too late for the present on-going feasibility study. However it would be useful to fine tune specific metallurgical circuits within the present and future mill complexes.

The Rosika deposit has been adequately drilled off to a depth of approximately 400 to 450 m below the surface. No further drilling is recommended on the Rosika deposit at this time.

The Coacia and Pisolita deposits have been drilled off to a depth of approximately 250 and 200 m below the surface, respectively. It is recommended that further drilling at the Coacia and Pisolita deposits be completed to target areas where there is wide-spaced drilling and Inferred Mineral Resources within their respective pit boundaries, requiring approximately 60 diamond drill holes totalling 12,000 m. The anticipated budget would be approximately US\$ 2,400,000 at an “all in” cost rate of US\$ 200 per metre.

A series of geotechnical drill holes are scheduled to be completed on the Incredible 6 deposits. This should be carried out as soon as practical so that the best optimized pit designs can be finalized. No further exploration holes are warranted on the Incredible 6 deposits at this time.

26.2 GEOTECHNICAL

Recommendations following the 2008/2009 TSF geotechnical site investigation included:

- Collecting and testing additional “undisturbed” overburden samples
- Additional testing of in situ overburden

- Additional testing of bedrock samples, and
- Additional testing of select overburden and bedrock samples to confirm laboratory results.

A number of triaxial tests of “undisturbed” samples were not completed due to insufficient sample size or unacceptable disturbance during sample extrusion. Four Shelby tube (ST) samples were not slated for testing and it is recommended that these samples be considered for triaxial testing. Although a maximum of four ST samples remain to be tested, sample availability, storage conditions and variability in the results may necessitate collection of additional ST samples for triaxial testing. It is also recommended that block samples be collected for direct shear testing.

Another method of determining the in situ overburden shear strength is to perform vane shear tests. It is recommended that vane shear testing be incorporated into any additional geotechnical drilling to provide additional information regarding the overburden in situ shear strength parameters.

The UCS test of a bedrock (gabbro) sample indicated that the tested UCS value was lower than the estimated UCS value. It is recommended that additional UCS testing be completed to confirm this result and, if necessary, adjust the estimated UCS values.

Additional laboratory testing of select overburden and bedrock samples should be conducted at a laboratory with well-known quality assurance and control standards to confirm the results from the GMK and UCAB laboratories. Select duplicate overburden samples were prepared for this purpose following the site investigation program and were being stored on site. The availability and condition of these samples should be confirmed.

Recommendations for open pit geotechnical:

- Open pit geotechnical drilling program for feasibility level design
- Mill site and waste dump geotechnical site investigation
- Feasibility level open pit slope design
- Feasibility level mill site and waste dump geotechnical design.

26.3 MINING - WASTE STORAGE

Waste rock dump seepage is monitored, but not collected. A seepage collection system is not planned for future dumps. It is recommended that PMG:

- Continues to monitor seepage and make modifications to future dump expansions and new waste rock dumps to allow for a collection and recovery system if seepage quality starts to degrade.
- Carry out a geotechnical assessment of waste dump foundations and stability.

26.4 METALLURGY

PRA concluded that, in general, 48-h CIL Au extractions of $\geq 90\%$ can be expected at a P_{80} of $\sim 74 \mu\text{m}$, 40% solids, $\leq 0.5 \text{ g/L}$ NaCN and pH ~ 11 , with typical consumptions of 0.6 kg/t NaCN and $< 1 \text{ kg/t}$ lime. Recommendations included:

- Careful metallurgical control due to head grade variability and different leach train processing required to optimize gold recovery.
- Tracking of CN-soluble gold for resource modeling purposes is recommended, as a basis for blending a more predictable feed.

26.5 SCHEDULE

The implementation schedule is based on delivery of the SAG and ball mills 55 weeks from the date of placing the order, assumed to be 18 weeks after Board approval to proceed. The scheduled mill installation and commissioning period is 37 weeks. This schedule should be updated to reflect PMG's approval date for project implementation and to capture changes in delivery periods for capital equipment, including long lead items such as gyratory crusher, SAG and ball mills and transformers. The mills are likely to be the critical item of equipment required to complete the project and should, therefore, be tendered and a recommendation to purchase issued as soon as practical after the commencement of sufficient engineering.

Other long-lead items such as transformers and the primary crusher will also need to be ordered near the commencement of engineering.

26.6 POWER SUPPLY

The source of permanent site power for the expansion has not been definitively resolved for this study, and the schedule impact of the power supply options is uncertain. Supply of permanent power for the expansion has the potential to become the schedule critical path.

26.7 CAPITAL ESTIMATE

The capital estimates presented in this section have been compiled by Micon from information provided by Ausenco and Knight Piésold during 2010. In Micon's opinion, the impact on these estimates of inflation and exchange rate variances between the effective date and the signature date of this report lie within the expected range of accuracy of the estimate.

However, due to the passage of time since the preparation of the capital and operating cost estimates used in this Feasibility Study, it is recommended that, subject to project approval, an update of the estimate should be prepared as a basis for management of the implementation phase.

26.8 BUDGET

The budget to carry out further technical work is shown in Table 26.1.

Table 26.1
Project Development Budget to March 2013

Item	Quantity	Rate	Cost (\$)
<i>Exploration/resource upgrades</i>			
Diamond drilling	25,000 metres	\$173/m	4,300,000
Assaying	20,000 samples	\$30/sample	600,000
Geology and support			100,000
Environmental and permitting work			50,000
Updated resource models			50,000
Subtotal			5,100,000
Contingency		10%	500,000
Total			5,600,000
<i>Design Work</i>			
Engineering; Plant			5,000,000
Engineering; Tailings			800,000
Environmental and permitting work			200,000
Geotechnical Studies			100,000
Metallurgical testwork	100 samples	\$500/sample	50,000
Travel and other support			50,000
Report preparation			70,000
Subtotal			6,270,000
Contingency		10%	630,000
Total			6,900,000
Grand total			12,500,000

27.0 DATE AND SIGNATURE PAGE

The effective date of the RCP, VBK and Incredible 6 deposit mineral resource estimates is as of 31 December, 2009. The effective date of the mineral reserve estimate and this report is 30 December, 2010.

Signed:

David Kent Makepeace, M.Eng., P.Eng.
Senior Geologist, Micon International Limited
Date: 30 December, 2011

Daniel Friedman, P.Eng.
Senior Engineer, Knight Piésold Ltd.
Date: 30 December, 2011

Dayan Anderson, QP, MMSA
Senior Mining Engineer, Micon International Limited
Date: 30 December, 2011

Richard Gowans, P.Eng.
Senior Metallurgist, Micon International Limited
Date: 30 December, 2011

Greg Lane, F.AusIMM.
General Manager, Technical Solutions, Ausenco
Date: 30 December, 2011

Christopher Jacobs, CEng MIMMM
Senior Consultant, Mineral Economics, Micon International Limited
Date: 30 December, 2011

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29.0 CERTIFICATES

CERTIFICATE OF AUTHOR

David K. Makepeace

As an author of this report entitled “Rusoro Mining Ltd., NI 43-101 Technical Report, Feasibility Study, Expansion of Gold Production at Choco 10 and Incredible 6, Bolivar State, Venezuela” dated December 30, 2011, I, David Makepeace, M.Eng., P.Eng., do hereby certify that:

1. I am employed by and carried out this assignment for:
Micon International Limited, Suite 205 – 700 West Pender Street,
Vancouver, British Columbia, V6C 1G8, Canada.
Telephone : (604) 647-6463, Fax : (604) 647-6455.
2. I hold the following academic qualifications:
 - Bachelor of Applied Science - Geological Engineering, Queen’s University at Kingston, Ontario, 1976,
 - Master of Engineering - Environmental Engineering, University of Alberta, 1994.
3. I am a registered member of the:
 - Association of Professional Engineers and Geoscientists of British Columbia.
4. I have worked as a geological engineer for a total of 32 years since my graduation from university.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for this report for the purposes of NI 43-101. My relevant experience in gold deposits includes mineral exploration, geological modeling, mineral resource estimates and operations of numerous properties in British Columbia, California, Montana, Mexico, Mali West Africa.
6. I am responsible for the preparation of Sections 4 to 12, 14 and 23 of this technical report.
7. I visited the property from July 29 to August 9, 2009.
8. I was the author of a previous report entitled “Technical Report on the Mineral Resources of the Choco 10 Deposits, Bolivar State, Venezuela” dated August 18, 2010.
9. As of the effective date of the report, to the best of my knowledge, information and belief, the sections of the Technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading. Mining has continued since the effective date.
10. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101 other than providing consulting services.
11. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

Dated at Vancouver, B.C. this 30th day of December, 2011.

(signed by) “David K. Makepeace”

David K. Makepeace, M.Eng., P.Eng.

(Sealed)

Professional Engineering Stamp

CERTIFICATE OF AUTHOR

Daniel Friedman

As an author of this report entitled “Rusoro Mining Ltd., NI 43-101 Technical Report, Feasibility Study, Expansion of Gold Production at Choco 10 and Increible 6, Bolivar State, Venezuela” dated December 30, 2011, I, Daniel Friedman, P.Eng., do hereby certify that:

1. I am employed by and carried out this assignment for:
Knight Piésold Ltd., Suite 1400 – 750 West Pender Street,
Vancouver, British Columbia, V6C 1T8, Canada.
Telephone : (604) 685-0543, Email: dfriedman@knightpiesold.com
2. I am a graduate of McGill University, Montreal, Canada (B.Eng. - Civil, 2003).
3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #32571).
4. I have practiced my profession continuously since June 2004. My relevant experience with respect to mine waste and water management includes over seven years of continuous work in the discipline.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for this report for the purposes of NI 43-101.
6. I am responsible for the preparation of Section 18.10 of this technical report.
7. I visited the property from November 26th to December 4th, 2008.
8. I have had no prior involvement with the properties that are the subject of the Technical Report.
9. As of the effective date of the report, to the best of my knowledge, information and belief, the sections of the Technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading. Mining has continued since the effective date.
10. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101 other than providing consulting services.
11. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

Dated at Vancouver, B.C. this 30th day of December, 2011.

(signed by) “Daniel Friedman”

Daniel Friedman, P.Eng.

(Sealed)

Professional Engineering Stamp

CERTIFICATE OF AUTHOR

Dayan Anderson

As the author of portions of the technical report entitled “NI 43-101 Technical Report, Feasibility Study, Expansion of Gold Production at Choco 10 and Increible 6, Bolivar State, Venezuela” dated December 30, 2011 (“Technical Report”) I, Dayan Anderson, do hereby certify that:

1. I am the Principal of Onyx Mining Services, California, and I carried out this assignment as an associate for Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario, M5H 2Y2- tel. (416) 362-5135, fax (416) 362-5763
2. I hold the following academic qualifications:
B.S. Mining Engineering, University of California, (1994).
3. I am a Qualified Professional (QP) Member of the Mining & Metallurgical Society of America (MMSA);
4. I have worked as a mining engineer in the minerals industry for over 17 years;
5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 13 years as a mining engineer at various active mine operations and four years with Micon as a senior mining engineering consultant;
6. I have not visited the Choco or Increible properties.
7. I am responsible for the preparation of Sections 15, 16 and 21.2.1 of this Technical Report.
8. I am independent of the issuer to whom this report is addressed, as described in Section 1.5 of NI 43-101;
9. I have had no prior involvement with the properties that are the subject of the Technical Report.
10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective Date: December 31, 2010

Signing Date: December 30, 2011

“Dayan Anderson” {signed}

Dayan Anderson, QP, MMSA

CERTIFICATE OF AUTHOR

Richard M. Gowans

As the author of portions of the technical report entitled “NI 43-101 Technical Report, Feasibility Study, Expansion of Gold Production at Choco 10 and Incredible 6, Bolivar State, Venezuela” dated December 30, 2011 (“Technical Report”) I Richard M. Gowans, P. Eng., do hereby certify that:

1. I am employed by, and carried out this assignment for:

Micon International Limited
Suite 900, 390 Bay Street
Toronto, Ontario,
M5H 2Y2
tel. (416) 362-5135 fax (416) 362-5763
e-mail: rgowans@micon-international.com

2. I hold the following academic qualifications:

B.Sc. (Hons) Minerals Engineering, The University of Birmingham, U.K., 1980

3. I am a registered Professional Engineer of Ontario (membership number 90529389); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum.
4. I have worked as an extractive metallurgist in the minerals industry for over 30 years.
5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the management of technical studies and design of numerous metallurgical testwork programs and metallurgical processing plants.
6. I have not visited the Choco or Incredible properties.
7. I supervised the preparation of, and am responsible for Sections 13 and 21, excluding sub-section 21.2.1, of this Technical Report.
8. I am independent of the issuer to whom this report is addressed, as described in Section 1.5 of NI 43-101.
9. I have had no prior involvement with the properties that are the subject of the Technical Report.
10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective Date: December 31, 2010

Signing Date December 30, 2011

“Richard M. Gowans” {signed and sealed}

Richard M. Gowans, P.Eng.

CERTIFICATE OF AUTHOR

Gregory Searle. Lane

As the author of portions of the technical report entitled “NI 43-101 Technical Report, Feasibility Study, Expansion of Gold Production at Choco 10 and Increible 6, Bolivar State, Venezuela” dated December 30, 2011 (“Technical Report”) I, Gregory Seale Lane, do hereby certify that:

1. I am General Manager Technical Solutions for Ausenco Minerals & Metals, 144 Montague Road, South Brisbane, Queensland, Australia
2. I graduated with a B. App. Sc. (Applied Chemistry) in 1982 and an M.Sc. degree in Chemistry from University of Tasmania in 1987
3. I am a Fellow of the Australian Institute of Mining and Metallurgy (F. AusIMM).
4. I have worked as a Metallurgist continuously since 1990. For the past 8 years I have been employed with Ausenco Minerals & Metals. During this period I have fulfilled roles as Manager Technology and GM Technical Solutions.
5. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purpose of this report.
6. I have not visited the Choco or Increible properties.
7. I am responsible for, Sections 17 and 18, excluding sub-section 18.10, of this Technical Report.
8. I am independent of the issuer to whom this report is addressed, as described in Section 1.5 of NI 43-101.
9. I have had no prior involvement with the properties that are the subject of the Technical Report.
10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective Date: December 31, 2010

Signing Date December 30, 2011

“Gregory Searle. Lane” {signed and sealed}

Gregory Searle Lane F.AusIMM
GM Technical Solutions
Ausenco Minerals Canada Inc.

CERTIFICATE OF AUTHOR

Christopher Jacobs

As the author of portions of the technical report entitled “NI 43-101 Technical Report, Feasibility Study, Expansion of Gold Production at Choco 10 and Increible 6, Bolivar State, Venezuela” dated December 30, 2011 (“Technical Report”) I, Christopher Jacobs, do hereby certify that:

1. I am employed by, and carried out this assignment for Micon International Limited, Suite 900 – 390 Bay St, Toronto, ON, M5H 2Y2 - tel. (416) 362-5135, fax (416) 362-5763
2. I hold the following academic qualifications:
B.Sc. (Hons) Geochemistry, University of Reading, 1980;
M.B.A., Gordon Institute of Business Science, University of Pretoria, 2004.
3. I am a Chartered Engineer registered with the Engineering Council of the U.K. (registration number 369178); Also, I am a professional member in good standing of the Institute of Materials, Minerals and Mining; and a member of the Canadian Institute of Mining, Metallurgy and Petroleum;
4. I have worked in the minerals industry for over 30 years;
5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 10 years as an exploration and mining geologist on gold, platinum, copper/nickel and chromite deposits; 10 years as a technical/operations manager in both open pit and underground mines; 3 years as strategic (mine) planning manager and the remainder as an independent consultant;
6. I visited the Choco and Increible properties during the period of October 19 to 24, 2009.
7. I am responsible for the preparation of Sections 1 to 3, 19, 20, 22, and 24 to 26 of this Technical Report.
8. I am independent of the issuer to whom this report is addressed, as described in Section 1.5 of NI 43-101;
9. I was co-author of a previous report entitled “Technical Report on the Preliminary Assessment of the Expansion of Production at Choco 10, Bolivar State, Venezuela”, dated June 3, 2009.
10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective Date: December 31, 2010

Signing Date: December 30, 2011

“Christopher Jacobs” {signed and sealed}

Christopher Jacobs, CEng, MIMMM