Amended NI 43-101 Technical Report Preliminary Economic Assessment Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana

Effective Date: July 8, 2015
Original Report Date: July 8, 2015
Amended Report Date: July 31, 2015

Report Prepared for

Luna Arena
Herikerbergweg 238
101 CM Amsterdam Zuidoost
The Netherlands

1090 Hamilton Street
Vancouver, BC V6B 2R9
Canada

Report Prepared by

SRK Consulting (U.S.), Inc.
7175 West Jefferson Avenue, Suite 3000
Lakewood, CO 80235

SRK Project Number: 417500.010

Signed by Qualified Persons:

Bart A. Stryhas, PhD, CPG
Bret C. Swanson, BEng Mining, MAusIMM, MMSAQP
Eric J. Olin, MSc Metallurgy, MBA, SME-RM, MAusIMM
Grant A. Malensek, MEng, PEng/PGeo
Mark A. Willow, MSc, CEM, SME-RM

Reviewed by:

Peter Clarke, BSc Mining, MBA, PEng

Table of Contents

1 Summary 1
1.1 Property Description, Location and Ownership 1
1.2 Geology and Mineralization 1
1.3 Status of Exploration, Development and Operations 1
1.4 Mineral Processing and Metallurgical Testing 2
1.5 Mineral Resource Estimate 3
1.6 Mineral Reserve Estimate 4
1.7 Mining Methods 4
1.8 Recovery Methods 6
1.9 Project Infrastructure 6
1.10 Environmental Studies and Permitting 7
1.11 Capital and Operating Costs 8
1.11.1 Mining Operating Costs 8
1.11.2 Process Operating Costs 9
1.11.3 Tailings Operating Costs 9
1.11.4 Support Operating Costs 10
1.11.5 Mining Capital Costs 10
1.11.6 Process Capital Costs 10
1.11.7 Infrastructure Capital Costs 11
1.11.8 Other Capital Costs 11
1.12 Economic Analysis 12
1.13 Conclusions and Recommendations 13
1.13.1 Geology and Resources 13
1.13.2 Open Pit Geotechnical Program 13
1.13.3 Mineral Processing and Metallurgical Testing 14
1.13.4 Mining 14
1.13.5 Recovery Methods 14
1.13.6 Tailings and Infrastructure 15
1.13.7 Environmental 15
1.13.8 Projected Economics 15
2 Introduction 17
2.1 Terms of Reference and Purpose of the Report 17
2.2 Qualifications of Consultants (SRK) 17
2.3 Details of Inspection 18
2.4 Sources of Information 18

2.5 Effective Date 18
2.6 Units of Measure 19
3 Reliance on Other Experts 20
4 Property Description and Location 21
4.1 Property Location 21
4.2 Mineral Titles 23
4.2.1 Geopolitical 23
4.2.2 Mineral Rights and Properties 25
4.2.3 Nature and Extent of Issuer's Interest 29
4.2.4 Location of Mineralization and Facilities 29
4.3 Royalties, Agreements and Encumbrances 31
4.4 Environmental Liabilities and Permitting 31
4.4.1 Environmental Liabilities 31
4.4.2 Required Permits and Status 31
4.5 Other Significant Factors and Risks 31
5 Accessibility, Climate, Local Resources, Infrastructure and Physiography 32
5.1 Accessibility 32
5.2 Climate 33
5.3 Resources and Infrastructure 33
5.4 Physiography 34
6 History 35
6.1 Prior Ownership and Exploration 35
6.2 Historical Mineral Resource Estimations 37
7 Geological Setting and Mineralization 38
7.1 Regional Geology 38
7.2 Property Geology 42
7.2.1 General 42
7.2.2 Lithology 43
7.2.3 Structure 47
7.3 Mineralization 48
8 Deposit Type 50
9 Exploration 51
10 Drilling 52
10.1 Guyanor Drilling Program: 1996 to 1998 52
10.2 Columbus Drilling Program: 2011 to 2012 54
10.3 Columbus Drilling Program: 2013 to 2014 55

10.4 Interpretation of Drillhole Results 58
11 Sample Preparation, Analysis and Security 60
11.1 Historical Methods 60
11.2 Columbus Drill Program 61
11.2.1 Core Logging and Sampling 61
11.2.2 Density Measurements 63
11.2.3 Sample Preparation and Analysis 64
11.3 Quality Assurance / Quality Control (QA/QC) 65
11.3.1 Conclusions 68
12 Data Verification 69
12.1 Procedures 69
12.2 Limitations 69
12.3 Opinion on Data Adequacy 69
13 Mineral Processing and Metallurgical Testing 70
13.1 Sample Compositing 70
13.2 Test Composite Characterization 74
13.2.1 Chemical Analyses 74
13.2.2 Mineralogical Analyses 76
13.2.3 Hardness Characterization 77
13.3 Metallurgical Testwork – Master Composites 78
13.3.1 Whole-Ore Cyanidation 78
13.3.2 Gravity Concentration + Cyanidation of Gravity Tailing 79
13.3.3 Gravity Concentration + Flotation 81
13.3.4 Rougher Flotation Concentrate Upgrading 85
13.3.5 Bulk Cleaner Concentrate Production 88
13.3.6 Concentrate Cyanidation 90
13.3.7 Overall Gravity + Flotation + Cyanidation Performance 91
13.4 Metallurgical Testwork: Variability Composites 92
13.4.1 Gravity + Cyanidation 92
13.4.2 Gravity + Flotation 93
13.5 Cyanide Detoxification 96
13.6 Thickening Studies 96
13.6.1 Flotation Tailings 97
13.6.2 Flotation Concentrate Leach Residues 97
13.7 Relevant Test Results 98
13.8 Gold Recovery Estimate Assumptions 101
13.9 Significant Factors 101

13.10 Process Option Trade-Off Analysis 101

14 Mineral Resource Estimate 104

14.1 Basis of Resource Estimation 104

14.2 General Geology 104

14.3 Controls on Gold Mineralization 104

14.4 Density 106

14.5 Sample Database 106

14.6 Capping and Compositing 106

14.7 Block Model 107

14.8 Estimation Strategy 107

14.9 Estimations Procedures 108

14.10 Model Validation 112

14.11 Resource Classification 113

14.12 Mineral Resource Statement 114

14.13 Mineral Resource Sensitivity 115

15 Mineral Reserve Estimate 117

16 Mining Methods 118

16.1 Introduction 118

16.2 Mine Block Model 119

16.3 Pit Optimization 122

16.4 Geotechnical 126

16.4.1 Pit Design Parameters 128

16.5 Pit Design 128

16.6 Grade Tonnage 130

16.7 Cut-Off Grade 133

16.8 Phase Design 133

16.9 Production Schedule 136

16.10 Overburden Storage 143

16.11 Haulage Profile 143

16.12 Fleet Estimate 146

16.12.1 Drilling 146

16.12.2 Load and Haul 147

16.12.3 Roads and Dumps 148

16.12.4 Fleet Estimate 148

16.13 Mine Operating Cost 149

16.14 Mine Capital Cost 153

17 Recovery Methods 154

17.1 Process Description 154
17.1.1 Run-of-Mine Pad 157
17.1.2 Crushing Circuit 157
17.1.3 Grinding, Classification and Gravity Circuit 158
17.1.4 Grinding Control Thickener 158
17.1.5 Leach and Carbon Adsorption Circuit 158
17.1.6 Elution and Gold Room Operations 159
17.1.7 Carbon Safety Screen 160
17.1.8 Cyanide Destruction Circuit 160
17.1.9 Tailings Disposal 161
17.2 Consumable Requirements 161
17.3 Metallurgical Accounting 162
17.4 Operating Cost Estimate 162
17.4.1 Labor 163
17.4.2 Consumables 165
17.4.3 Power 165
17.4.4 Maintenance Supplies 165
17.5 Capital Cost Estimate 165
17.6 Significant Factors 166
18 Project Infrastructure 167
18.1 Infrastructure and Logistic Requirements 167
18.1.1 On-Site Infrastructure 167
18.1.2 Site Water Management 167
18.1.3 Service Roads and Bridges 167
18.1.4 Mine Operations and Support Facilities 168
18.1.5 Process Support Facilities 169
18.1.6 Additional Support Facilities 169
18.1.7 Power Supply and Distribution 170
18.1.8 Water Supply 170
18.2 Tailings Management Area 170
18.3 Area Hydrology and Water Balance 175
19 Market Studies and Contracts 179
20 Environmental Studies, Permitting and Social or Community Impact 180
20.1 Site Visit 180
20.2 Current Liabilities 180
20.3 Required Permits and Status 180
20.3.1 Required Permits and Status 181

20.3.2 Facilities Classified for Environmental Protection (ICPE) 182
20.3.3 Restoration of the Access Road from the Croisée D'apatou 182
20.3.4 Law on Water and Aquatic Environments 183
20.3.5 European Directives 183
20.3.6 Mine Code Reformation 183
20.4 Environmental Study Results 184
20.4.1 Air Quality 184
20.4.2 Cultural and Archeological Resources 184
20.4.3 Biological Reserves and Resources 185
20.4.4 Threatened, Endangered, and Special Status Species 185
20.4.5 Land Use 186
20.4.6 Hydrogeology (Groundwater) 186
20.4.7 Geochemistry 186
20.5 Environmental and Social Management Planning 187
20.6 Reclamation and Closure 188
20.7 Socio-Economics 188
20.7.1 Gold Mining 188
20.7.2 Bushinengues 189
20.8 Environmental / Social Issues and Impacts 189
20.8.1 Principal Issues 189
20.8.2 Project Advantages 190
20.9 International Standards and Guidelines 190
21 Capital and Operating Costs 193
21.1 Capital Cost Estimates 193
21.1.1 Mine 193
21.1.2 Process Capital Costs 194
21.1.3 Infrastructure Capital Costs 195
21.1.4 Other Capital Costs 195
21.2 Operating Cost Estimates 195
21.2.1 Mine Operating Costs 196
21.2.2 Process Operating Costs 196
21.2.3 Support Operating Costs 197
22 Economic Analysis 198
22.1 Principal Assumptions and Input Parameters 198
22.2 Cashflow Forecasts and Annual Production Forecasts 198
22.3 Taxes, Royalties and Other Interests 202
22.3.1 Royalties 202

22.3.2 Income Taxes 202
22.4 Sensitivity Analysis 202
23 Adjacent Properties 206
24 Other Relevant Data and Information 207
24.1 Project Impacts on Economy 207
24.2 Employment Income Sources 210
24.3 Public Health and Safety 210
24.4 Long-Term Dependence and Sustainability 211
24.5 Cultural Property 211
25 Interpretation and Conclusions 212
25.1 Geology and Resources 212
25.2 Mining 212
25.3 Metallurgy and Processing 212
25.3.1 Recovery Methods 213
25.4 Environmental and Social 213
25.5 Projected Economic Outcomes 214
25.6 Foreseeable Impacts of Risks 214
26 Recommendations 216
26.1 Recommended Work Programs and Costs 216
26.1.1 Exploration Drilling 216
26.1.2 Open Pit Geotechnical Program 216
26.1.3 Mine Recommendations 217
26.1.4 Tailings and Infrastructure 217
26.1.5 Environmental 218
26.1.6 Economics 218
26.1.7 Costs 218
27 References 220
28 Glossary 222
28.1 Mineral Resources 222
28.2 Mineral Reserves 222
28.3 Definition of Terms 223
28.4 Abbreviations 224

List of Tables

Table 1.4.1: Summary of Estimated Gold Recoveries from Process Options Tested 2
Table 1.5.1: Montagne d'Or Mineral Resource Statement as of April 11, 2015 SRK Consulting (U.S.), Inc. 3

Table 1.11.1: Total Operating Cost Summary 8
Table 1.11.2: Total Capital Cost Summary (US$000's) 8
Table 1.11.1.1: Mine Operating Cost Summary 9
Table 1.11.2.1: Summary of Process Plant Operating Costs 9
Table 1.11.3.1: Summary of Tailings Operating Costs 9
Table 1.11.4.1: Site G&A Operating Costs 10
Table 1.11.5.1: Open Pit Mining Capital Costs (US$000's) 10
Table 1.11.6.1: Process Plant Capital Cost Estimate (US$000s) 11
Table 1.11.7.1: Infrastructure Capital Cost Estimate (US$000's) 11
Table 1.11.8.1: Owners and Closure Capital Cost Estimate (US$000s) 12
Table 1.12.1: LoM After-Tax Indicative Economic Results (US$000's) 12
Table 1.12.2: Sensitivity Analysis at Various Gold Price Points 13
Table 2.3.1: Site Visit Participants 18
Table 4.2.2.1: Land Tenure of the Paul Isnard Project 27
Table 6.2.1: Previous Resource Estimates for the Montagne d'Or Deposit 37
Table 10.1.1: Drillholes (56 in Total) Drilled by Guyanor from 1996 to 1998 53
Table 10.2.1: Drillholes (45 in total) completed by Columbus (Phase 1) in 2011 and 2012 55
Table 10.3.1: Drillholes (126 in total) completed by Columbus (Phase 2) in 2013 and 2014 56
Table 11.2.2.1: Listing of Montagne d'Or Prospect Dry Rock Density Measurements 64
Table 13.1.1: Drillholes and Intervals Used for the Metallurgical Program 71
Table 13.1.2: UFZ Master Compositing List 73
Table 13.1.3: UFZ Variability Compositing List 74
Table 13.1.4: LFZ Master Compositing List 74
Table 13.1.5: LFZ Variability Compositing List 74
Table 13.1.6: Saprolite Variability Compositing List 74
Table 13.2.1.1: Gold and Silver Analyses on UFZ and LFZ Master and Variability Composites 75
Table 13.2.1.2: Elemental Analyses on UFZ and LFZ Master and Variability Composites 76
Table 13.2.2.1: Mineral Percentages in UFZ and LFZ Master Composites 77
Table 13.2.3.1: Bond Ball Mill, Crushing and Abrasion Index Results 78
Table 13.2.3.2: Summary of SMC Evaluation 78
Table 13.3.1.1: Summary of Whole-Ore Cyanidation versus Grind Size 79
Table 13.3.2.1: Summary of Gravity + Cyanidation Tests Versus Grind Size 80
Table 13.3.2.2: Comparison of Whole-ore Cyanidation and Gravity + Cyanidation Results 80
Table 13.3.2.3: Cyanidation of Gravity Tailings versus Cyanide Concentration 81
Table 13.3.3.1: Gravity + Flotation Tests versus Grind Size on UFZ Master Composite 82
Table 13.3.3.2: Gravity + Flotation Tests versus Grind Size on the LFZ Master Composite 83
Table 13.3.3.3: Summary of Large Batch Gravity + Flotation Tests Results on the UFZ Master Composite 84

Table 13.3.3.4: Summary of Large Batch Gravity + Flotation Tests Results on the LFZ Master Composite ..85
Table 13.3.4.1: Cleaner Flotation Tests versus Reagent Dosage on UFZ Rougher Concentrate 86
Table 13.3.4.2: Cleaner Flotation Tests versus Reagent Dosage on LFZ Rougher Concentrate 86
Table 13.3.4.3: Summary of Cleaner Flotation Tests Vs Regrind Size 87
Table 13.3.5.1: Summary of Large Batch Gravity + Flotation Test Results on the UFZ and LFZ Master Composites 89
Table 13.3.6.1: Summary of Concentrate Cyanidation versus Cyanide Concentration 90
Table 13.3.7.1: Overall Gold Recoveries with Gravity + Flotation + Concentrate Cyanidation 92
Table 13.4.1.1: Summary of Gravity + Cyanidation Results on Variability Composites 93
Table 13.4.2.1: Summary of Gravity + Flotation Results on Variability Composites 95
Table 13.5.1: Summary of Cyanide Detoxification Tests Flotation Concentrate Leach Residues 96
Table 13.5.2: Cyanide Detoxification Reagent Consumptions 96
Table 13.7.1: Estimated Gold Recoveries from Relevant Tests Conducted to Evaluate Three Different Process Options 99
Table 13.7.2: Estimated Silver Recoveries 100
Table 13.8.1: Summary of Estimated Gold Recoveries from Process Options Tested 101
Table 13.10.1: Trade-off Analysis Assumptions 102
Table 13.10.2: Trade-off Analysis Incremental Discounted Cash Flow Results 103
Table 14.3.1: Average Orientations of Structural Fabrics 105
Table 14.4.1: Densities Assigned in the Block Model 106
Table 14.7.1: Block Model Size and Extents 107
Table 14.9.1: Au Grade Estimation Parameters 109
Table 14.10.1: Estimation Performance Parameters of Au Estimation in Grade Shell 112
Table 14.10.2: Model Validation Statistical Results in Grade Shell 112
Table 14.10.3: Model Validation nearest Neighbor Results in Grade Shell 112
Table 14.12.1: Montagne d'Or Mineral Resource Statement as of April 11, 2015 SRK Consulting (U.S.), Inc.115
Table 14.13.1 Mineral Resource Sensitivity (1) 115
Table 16.2.1: Mine Block Model Variables 120
Table 16.2.2: Block Model Scripts 121
Table 16.3.1: Pit Optimization Inputs 123
Table 16.4.1.1: Recommended Pit Design Parameters (Hard Rock) 128
Table 16.5.1: Pit Design Parameters 129
Table 16.6.1: Grade Tonnage Curve within the PEA Pit Design 130
Table 16.6.2: Grade Tonnage Curve within the Maximum IRR Pit Design 132
Table 16.7.1: Cut-Off Calculations and Grade Bins 133
Table 16.8.1: Phase Inventory for Production Schedule 135
Table 16.9.1: CPLEX Optimization Limits 136
Table 16.9.2: Stockpile Inventory Schedule 137

Table 16.10.1: Overburden Storage Areas....143
Table 16.11.1: Rimpull Curve Representing Truck Speeds by Gradient for 777 Truck....144
Table 16.11.2: Annual Production and Cycle Time Information that Form Basis of Fleet Estimate....145
Table 16.12.1: Operating Hour Assumption for Major Fleet Items....146
Table 16.12.1.1: Blasthole Design Criteria....147
Table 16.12.2.1: Loading Design Criteria....148
Table 16.12.4.1: Suggested Mine Equipment Fleet....149
Table 16.13.1: Equipment Assumptions....149
Table 16.13.2: Mine Operating Cost....150
Table 16.13.3: Mine Labor Cost....151
Table 16.13.4: Annual Operating Cost....152
Table 16.14.1: Mine Capital and Sustaining Capital Conditions....153
Table 16.14.2: Initial and Sustaining Capital Estimate....153
Table 17.1.1: Preliminary Design Criteria for the Montagne d'Or Process Plant....156
Table 17.1.2: Process Plant Major Equipment List....157
Table 17.4.1: Summary of Process Plant Operating Costs....162
Table 17.4.1.1: Process Plant Manpower Schedule and Estimated Labor Cost....164
Table 17.4.2.1: Estimated Reagent Consumption and Cost....165
Table 17.5.1: Process Plant Capital Cost Estimate (US$000s)....166
Table 18.2.1 Tailings Dam Physicals....172
Table 18.2.2: TSF Capital and Sustaining Cost Estimate (US$)....174
Table 18.3.1: Annual Tailings Water Balance by Phase....176
Table 20.9.1: IFC Performance Standard vs. Compliance Approach....192
Table 21.1.1: Life-of-Mine Capital Costs (US$000's)....193
Table 21.1.1.1: Open Pit Mining Capital Costs (US$000's)....193
Table 21.1.1.2: Open Pit Mine Capital Costs....194
Table 21.1.2.1: Process Plant Capital Costs (US$000's)....194
Table 21.1.2.2: Tailings Storage Facility Capital Costs (US$000's)....194
Table 21.1.3.1: Infrastructure Capital Costs (US$000's)....195
Table 21.1.4.1: Owner's Costs (US$000's)....195
Table 21.1.4.2: Closure/Reclamation Capital Costs (US$000's)....195
Table 21.2.1: Life-of-Mine Operating Cost Summary....196
Table 21.2.1.1: Mine Operating Cost Summary....196
Table 21.2.2.1: Process Plant Operating Costs....196
Table 21.2.2.2: Tailings Capital Costs....197
Table 21.2.3.1: Support Operating Costs....197
Table 22.1.1: Basic Model Parameters....198

Table 22.2.1: Life-of-Mine Production Summary 198
Table 22.2.2: Life-of-Mine Process Production Summary 199
Table 22.2.3: Life-of-Mine After-Tax Indicative Economic Results (in US$000's) 200
Table 22.2.4: Life-of-Mine All-In Sustaining Cost (AISC) Contribution 200
Table 22.2.5: Life-of-Mine Annual Cash Flow Forecast 201
Table 22.4.1: Sensitivity Analysis of After-Tax NPV 8% (US$ million) 202
Table 22.4.2: Sensitivity Analysis of After-Tax IRR 202
Table 22.4.3: Sensitivity Analysis at Various Gold Price Points 204
Table 24.1.1: Major Commodities and Direct Labor Estimate For the Project Life 208
Table 26.1.7.1: Summary of Costs for Recommended Work 219
Table 28.3.1: Definition of Terms 223
Table 28.4.1: Abbreviations 224

List of Figures

Figure 1.7.1: Life-of-Mine Production Schedule 5
Figure 1.7.2: Saprolite versus Hard Rock Production Schedule 5
Figure 1.9.1: Near Mine Site Infrastructure 7
Figure 4.1.1: Paul Isnard Project General Location Map 22
Figure 4.2.1.1: Location of the Potential Resource Pit Outline and Biological Reserve 25
Figure 4.2.2.1: Location of Columbus Concessions and PER Applications 28
Figure 4.2.4.1: Paul Isnard Project General Site Map 30
Figure 5.1.1: Picture of Helipad/airstrip at Camp Citron 33
Figure 6.1.1: Plan Map Overview of Historic Exploration Campaigns 37
Figure 7.1.1: Large Scale Geological Map of French Guiana 39
Figure 7.1.2: Large Scale Overview of the Geology of Northern French Guiana, showing the location of the Paul Isnard Project 40
Figure 7.1.3: Map Showing Correlation of the Guiana Shield with the West African Birimian Shield 42
Figure 7.2.2.1: Schematic Overview of Main Local Geological Units 44
Figure 7.3.1: Example of Visible Gold Occurring within Mafic Volcanics (MO1266) 49
Figure 10.4.1: Plan View of Drillhole Traces 59
Figure 11.2.1.1: Core Logging Facility at the Citron Exploration Camp 62
Figure 11.2.1.2: Core Racks at the Citron Exploration Camp 63
Figure 11.3.1: Results of Au Standard at 0.599 g/t 66
Figure 11.3.2: Results of Au Standard at 1.807 g/t 66
Figure 11.3.3: Results of Au Standard at 5.96 g/t 67
Figure 11.3.4: Results of all Blank Analyses 67
Figure 13.1.1: Metallurgical Drillhole Locations 72

Figure 13.3.1.1: Gold Extraction versus Leach Retention Time.....79
Figure 13.3.2.1: Gold Extraction from Gravity Tailings versus Retention Time.....81
Figure 13.3.6.1: Gold Extraction versus Leach Retention Time.....91
Figure 13.3.6.2: Silver Extraction versus Leach Retention Time.....91
Figure 13.4.1.1: Gold Extractions versus Leach Retention Time.....93
Figure 14.3.1: Box Plot of Gold Grade by Lithology.....105
Figure 14.6.1: Log Normal Cumulative Distribution Plot of Gold Assays above 10 g/t.....107
Figure 14.9.1: Representative Cross Section 173,000E with Estimated Au Grades.....110
Figure 14.9.2: Representative Cross Section 174,000E with Estimated Au Grades.....111
Figure 14.10.1 North-South Oriented Swath Plots.....113
Figure 14.11.1: Representative Cross Section of Resource Classification.....114
Figure 14.13.1: Sensitivity of Tonnes and Grade to Cut-off.....116
Figure 16.1.1 Montagne d'Or Site Layout.....118
Figure16.3.1: Pit by Pit Analysis.....124
Figure 16.3.2: Optimization Analysis.....125
Figure 16.3.3: Pit Shells Compared to Resource Potential.....125
Figure 16.4.1: Empirical Design Chart for MRMR versus Slope Height.....127
Figure 16.5.1: PEA Pit.....129
Figure 16.6.1: Grade Tonnage Curve within PEA Pit.....131
Figure 16.6.2: Grade Tonnage Curve within Maximum IRR Pit.....132
Figure 16.8.1: Phase Layout.....134
Figure 16.9.1: Grade Bin Schedule.....138
Figure 16.9.2: Saprolite, Hard Rock and RoM Production Schedule.....139
Figure 16.9.3: Pre-Production Stripping.....139
Figure 16.9.4: Year 1 Pit Phase Plot.....140
Figure 16.9.5: Year 2 Pit Phase Plot.....140
Figure 16.9.6: Year 3 Pit Phase Plot.....140
Figure 16.9.7: Year 4 Pit Phase Plot.....141
Figure 16.9.8: Year 5 Pit Phase Plot.....141
Figure 16.9.9: Year 6 Pit Phase Plot.....141
Figure 16.9.10: Year 7 Pit Phase Plot.....142
Figure 16.9.11: Year 8 Pit Phase Plot.....142
Figure 16.9.12: Year 9 Pit Phase Plot.....142
Figure 16.9.13: Year 10 Pit Phase Plot.....143
Figure 16.11.1: Cross-Section of Haulage Cycle Time.....145
Figure 17.1.1: Conceptual Process Flowsheet.....155
Figure 18.2.1 Tailings Storage Facility Options.....171

Figure 18.2.2 TSF Tailings Dam and Site Layout 173
Figure 18.2.3 Embankment Height versus Storage Capacity Curve 175
Figure 18.3.1: WSP Water Sampling Locations 177
Figure 22.4.1: Project NPV 8% Sensitivities (US$ million) 203
Figure 22.4.2: Project IRR Sensitivities 203
Figure 22.4.3: Project NPV 8% Sensitivities at Varying Gold Prices 204
Figure 22.4.4: Project NPV Sensitivities at Varying Discount Rates 205

Appendices

Appendix A: Certificates of Qualified Persons

1 Summary

This report was prepared as a National Instrument 43-101 Technical Report (Technical Report), Preliminary Economic Assessment (PEA) for Nord Gold N.V. (Nordgold) with Columbus Gold Corporation (Columbus) by SRK Consulting (U.S.), Inc. (SRK) on the Montagne d'Or Gold Deposit (Montagne d'Or or Project) located in French Guiana. Columbus is the Project owner/operator and is currently exploring the deposit under an option agreement with Nordgold.

1.1 Property Description, Location and Ownership

Montagne d'Or is part of the larger Paul Isnard Project. The Project consists of eight mining concessions and two pending exploration permit applications covering a total area of 190 km^2. The Project area and mining concessions are located in the northwestern portion of French Guiana, South America. The Project area extends from longitude 53° 53' 52'' W (UTM 178,475) to 54° 03' 09'' W (UTM 161,360), and latitude 4° 40' 59'' N (UTM 518,322) to 4° 51' 03'' N (UTM 536,922). The Project also includes historic artisanal mining operations, exploration roads, drill pads, a core logging/storage facility and a base camp. The Camp Citron base camp is located approximately 4 km northwest of the deposit. Columbus is the Project owner/operator and is currently exploring the deposit under an option agreement with Nordgold.

1.2 Geology and Mineralization

The Montagne d'Or deposit is an Archean age, VMS gold deposit that has undergone remobilization and shear zone style deformation. The deposit is located within the northern greenstone belt of the Guiana Shield in French Guiana. Mineralization is hosted within the two billion year old, Paramaca Formation composed predominantly of meta-volcanic and meta-sedimentary units. These units have been deformed by ductile deformation resulting in tight to isoclinal folding and shearing as well as a pervasive foliation striking east-west and dipping steeply to the south. The current model of gold mineralization is a high sulfidization, volcanogenic type. Significant portions are thought to have been emplaced as replacement style mineralization. Subsequently, the mineralization has been deformed and partly remobilized within structural controls. Gold mineralization is associated with primary sulfide minerals as replacements within pyrite and chalcopyrite. At a macroscopic scale, the following five types of mineralization have been identified in mapping and drill core logging:

- Semi-massive sulfides (SMS, >20% sulfides) with associated gold mineralization;
- Sulfides as disseminations and stringers with associated gold mineralization;
- Late-stage disseminated euhedral pyrite mineralization;
- Rhythmic mafic tuff with associated pyrrhotite mineralization; and
- Gold mineralization associated with quartz veins.

1.3 Status of Exploration, Development and Operations

The database supporting the resource estimation of this report is current to April 11, 2015. It contains information from 224 diamond drillholes and 37 channel samples. The drilling was completed in two main campaigns. A previous owner drilled 56 holes between 1996 and 1998. Columbus completed an additional 171 holes from 2011 to November, 2014. The channel samples were all collected from surface outcrops between 1995 and 1997. SRK has previously reviewed the 1995 through 1998 exploration data and found it to be of sufficient quality to support an industry standard, resource estimation. All drilling, sampling and analytical work conducted by Columbus has followed industry standard procedures and includes quality assurance/quality control (QA/QC) protocols.

1.4 Mineral Processing and Metallurgical Testing

Bureau Veritas Commodities Canada Ltd. - Inspectorate Metallurgical Division (Inspectorate) was retained by Nordgold to perform metallurgical testing on samples from the Project located in north-west French Guiana. The test program was directed and supervised by Eric Olin from SRK. The results of this metallurgical investigation are fully documented in Inspectorate's report, "Metallurgical Testing to Recover Gold and Silver from the Montagne d'Or Gold Project, French Guiana," dated March 30, 2015.

The test program was focused on the testing of two master composites formulated from available whole core intervals representing the Upper Felsic Zone (UFZ) and the Lower Favorable Zone (LFZ), as well as selected variability composites.

Three process options, including whole-ore cyanidation, a combination of gravity concentration followed by cyanidation of gravity tailing, and gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate, were investigated on two master composites, and the preferred process option and optimal conditions were further verified on ten variability test composites.

Table 1.4.1 provides a summary of estimated gold recoveries achievable by each of the process options tested. Gold recovery achievable by a process flowsheet that includes gravity concentration followed by cyanidation is estimated at 95% from the UFZ and LFZ zones and 94% from the saprolite zones.

Gold recovery from a process flowsheet that includes gravity concentration followed by gold flotation from the gravity tailings and cyanide leaching of the flotation concentrate is estimated at 90% for the UFZ and LFZ zones and 65% for the saprolite zones. Estimated gold recoveries have been reduced by a 2% adjustment factor to allow for gold and silver losses that will occur during commercial operation due to plant inefficiencies.

Table 1.4.1: Summary of Estimated Gold Recoveries from Process Options Tested

Process Option	Calc. Head Au, g/t	Au Extraction %	Adjustment Factor	Au Recovery %
Whole-ore Cyanidation				
UFZ Master Composite	1.42	95	2	93
LFZ Master Composite	2.17	95	2	93
Gravity + Cyanidation				
UFZ Master Composite	1.79	97	2	95
LFZ Master Composite	1.80	97	2	95
Variability Composite (Average)	2.13	96	2	**94**
Saprolite	0.97	96	2	**94**
Gravity + Flotation + Cyanidation				
UFZ Master Composite	1.75	91	2	89
LFZ Master Composite	1.78	93	2	91
Variability Composite (Average)	1.98	90	2	**88**
Saprolite	0.69	67	2	**65**

Source: SRK, 2015

1.5 Mineral Resource Estimate

Gold mineralization is controlled mainly by structural fabric and lithology. The mineralization is localized in planar zones which have recurrent distribution and highly variable grades. Anomalous gold grades typically occur in zones 3 to 10 m wide which are separated by barren or lower grade zones 10 to 30 m wide. As part of the most recent drilling campaign, most of the historic core was re-logged to create a unified system of lithologic descriptions. This has resulted in a detailed, 3-D geologic model created by using Leapfrog® Geo software. Lithologic control of mineralization is evident and SRK utilized four lithic types or groups which were estimated independently.

The Au capping level was chosen at 39 g/t resulted in 25 samples ranging from 40.1 to 163 g/t being reduced to 39 g/t prior to compositing. This capping results in a net loss of 3% of all gold in the database. Compositing was completed in 3 m downhole lengths with no breaks at lithologic contacts.

Columbus constructed Leapfrog® software generated wireframe solids which enclose anomalous gold mineralization at a 0.3 g/t Au threshold. The grade estimation was conducted in eight domains. Four rock types/groups were used and each rock type/group was estimated independently both internal and external to the grade shell using only samples from the same domain. An Inverse Distance Weighting Squared (IDW^2) algorithm was used for the grade estimations.

Five techniques were used to evaluate the validity of the block model including; visual checks, overall model performance parameters, statistical comparison between composite and block grades, nearest neighbor comparisons and swath plots.

The Mineral Resources reported for the Montagne d'Or deposit are classified as Indicated and Inferred Mineral Resources, based primarily on drillhole spacing since all other supporting data is of good quality. Wire frame solids were constructed around the areas where the average drillhole spacing is approximately 50 m or less and these were used to assign the Indicated Mineral Resource classification. All blocks outside of these wireframes were classified as Inferred Mineral Resources.

The Montagne d'Or Mineral Resource statement is presented in Table 1.5.1. The resource is confined within a Whittle™ optimization pit shell and a cut-off grade (CoG) of 0.4 g/t Au applied. The pit shell and CoG assumes open-pit mining methods and is based on a mining cost of US$1.50/t, milling cost of US$15/t, administration cost of US$1/t, a gold price of US$1,300/oz., 90% gold recovery, gold refining cost of US$8/oz, and 5% net smelter return (NSR) royalty. A 45° pit shell slope was used for bedrock and a 35° pit shell slope was used for saprolite. The reported Mineral Resources include material from all estimation domains.

Table 1.5.1: Montagne d'Or Mineral Resource Statement as of April 11, 2015 SRK Consulting (U.S.), Inc.

Classification	Au Cut-Off (g/t)	Tonnes (M)	Au (g/t)	Contained Au (M oz)
Indicated	0.40	83.24	1.455	3.893
Inferred	0.40	22.37	1.550	1.115

- **Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.**
- All figures rounded to reflect the relative accuracy of the estimates.
- Metal assays were capped where appropriate.
- Mineral Resources are reported based on a CoG of 0.4 g/t Au, and are reported inside a conceptual pit shell based on appropriate mining and processing costs and metal recoveries for oxide and sulfide material.
- CoGs are based on a mining cost of US$1.50/t, milling cost of US$15/t, administration cost of US$1/t, a gold price of US$1,300/oz., 90% gold recovery, gold refining cost of US$8/oz, and 5% NSR royalty.

Source: SRK, 2015

1.6 Mineral Reserve Estimate

No Mineral Reserves are reported for a PEA.

1.7 Mining Methods

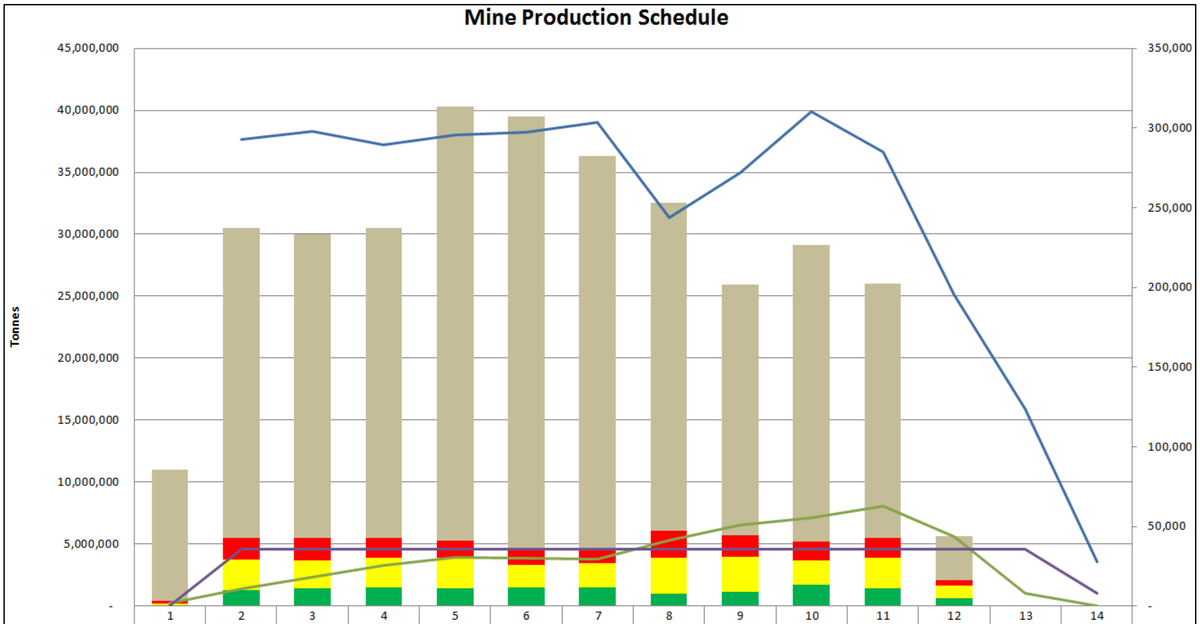
Montagne d'Or in French Guiana is located on the side of a moderately sized hill, surrounded by dense tropical rainforest in a remote location that has been disturbed by garimpeiro miners. Recent exploration programs have successfully confirmed mineralization along strike of the deposit resulting in approximately 5 Moz of gold potentially available for extraction. French Guiana is considered an extension of mainland France and as such SRK used labor rates consistent with those of developed countries for capital and operating cost estimation.

The open pit mining operation envisaged for Montagne d'Or will comprise traditional open pit mining equipment utilizing correctly sized loaders and two sizes of mining trucks. The operation is sized to produce 12,500 t/d of mill feed with a low grade stockpile to ensure high grade mill feed is processed first. The mine plan utilizes a phase bench sequence approach that follows precedence relationships, maintains a reasonable balanced fleet, provides approximately 300 koz of gold per year at a mining cost of US$2.37/t or US$815 million for the 11 years of full mine production. To achieve this, mine capital is estimated at US$86 million over the life-of-mine (LoM) and US$54 million initially.

The PEA open pit is approximately 2.5 km long by 500 m wide and 400 m deep with a total volume of 127.7 Mm^3 with a stripping ratio of 5 t of waste for every tonne of mill feed.

Although the Montagne d'Or project is a PEA and the inclusion of Inferred material is permitted, it should be noted that there is only 6% Inferred material contained within the PEA pit being sent to the mill or alternately 3.2 Mt Mill feed above a 0.7 g/t Au CoG.

Figure 1.7.1 illustrates the LoM production schedule broken out by grade bin tonnages corresponding to a gold cut-off of 0.7 g/t that is equivalent to US$1,200/Au oz (Low-Grade, LG), 1.0 g/t that is equivalent to US$800/Au oz (Medium-Grade, MG) and 2.1 g/t that is equivalent to US$400/Au oz (High-Grade, HG).

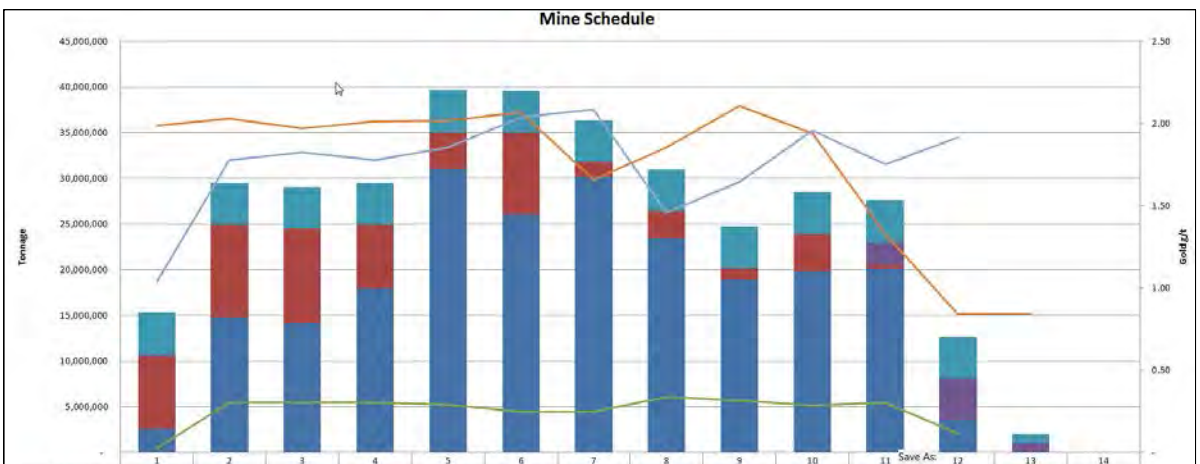


	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Waste	10,550,00	25,000,00	24,503,64	24,981,84	35,000,00	35,000,00	31,799,08	26,438,09	20,209,00	23,933,69	20,523,04	3,559,789		
LG Tonnes	237,051	1,714,734	1,818,372	1,617,337	1,380,166	1,173,414	1,013,796	2,184,973	1,764,468	1,539,769	1,580,684	460,975		
MG Tonnes	209,760	2,510,635	2,231,756	2,376,727	2,440,960	1,847,411	1,948,490	2,923,365	2,838,651	1,938,744	2,460,540	958,572		
HG Tonnes	3,189	1,274,632	1,449,873	1,505,936	1,455,241	1,479,175	1,537,714	970,369	1,135,571	1,707,499	1,458,776	658,022		
LG Stock	237,051	1,375,000	2,312,500	3,250,000	3,963,867	3,888,867	3,826,367	5,342,573	6,518,763	7,129,775	8,067,275	5,582,344	1,019,844	-
Plant Feed	-	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	1,019,844
Mill Au Ounces		292,544	297,857	289,522	295,765	297,068	303,451	243,743	272,139	310,100	284,747	195,375	123,685	27,647

Source: SRK, 2015

Figure 1.7.1: Life-of-Mine Production Schedule

Figure 1.7.2 illustrates the LoM production schedule showing the breakout of saprolite rock versus hard rock and the mined grade versus mill grade after stockpiling.



	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Plant Feed	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	1,019,844	
Rehandle	212,949	-	-	-	75,000	62,500	-	-	-	-	2,484,931	4,562,500	1,019,844	
Waste Saprolite	7,958,204	10,156,603	10,353,370	7,035,209	3,918,008	8,904,764	1,571,898	2,941,306	1,323,676	4,110,951	370,556	-	-	-
Waste Tonnes	2,591,796	14,843,397	14,150,278	17,946,638	31,081,992	26,095,236	30,227,190	23,496,786	18,885,329	19,822,745	20,152,486	3,559,789	-	-
Rom Mill Tonnes	450,000	5,500,000	5,500,000	5,500,000	5,276,367	4,500,000	4,500,000	6,078,707	5,738,690	5,186,012	5,500,000	2,077,569		
Plant Feed Au	1.99	2.03	1.97	2.02	2.02	2.07	1.66	1.86	2.11	1.94	1.33	0.84	0.84	
Rom Mill Grade	1.04	1.78	1.83	1.78	1.86	2.04	2.09	1.46	1.65	1.96	1.75	1.92		

Source: SRK, 2015

Figure 1.7.2: Saprolite versus Hard Rock Production Schedule

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.8 Recovery Methods

Metallurgical testwork was conducted to evaluate three different process flowsheet options including:

- Whole-ore cyanidation;
- Gravity concentration followed by cyanidation of the gravity tailing; and
- Gravity concentration followed by gold and silver flotation from the gravity tailing and cyanidation of the flotation concentrate.

After conducting a trade-off study the process flowsheet that includes gravity concentration followed by cyanidation of the gravity tailing was selected as this flowsheet option offers higher overall gold and silver recoveries and resulted in the highest Project net present value (NPV) and highest internal rate of return (IRR).

The selected process flowsheet will include gravity concentration followed by cyanidation of the gravity tailings to recover the contained gold and silver values, and will incorporate process unit operations that are standard to the industry, including: crushing, grinding, agitated cyanide leaching, gold and silver adsorption onto activated carbon, gold and silver desorption, electrowinning and refining. In addition, the cyanidation tailings will be detoxified to less than 1 ppm CN_{wad} with the well established INCO SO_2/air process.

1.9 Project Infrastructure

The major infrastructure items, such as the processing plant, overburden storage areas, and tailings storage facilities (TSF), have been conceptually located. Nordgold indicated that a sterilization drilling program will be carried out in 2015 to test these locations for suitability. Once the locations are found to be suitable, the process of detailed engineering can refine the supporting site infrastructure. SRK allocated US$84 million for site infrastructure not directly related to the process plant and mine equipment. This covers US$12.5 million for a potential water treatment plant, US$33 million for power generation, and US$25 million for road upgrades and other items, and a 20% contingency. Figure 1.9.1 illustrates the current Infrastructure layout near the open pit.

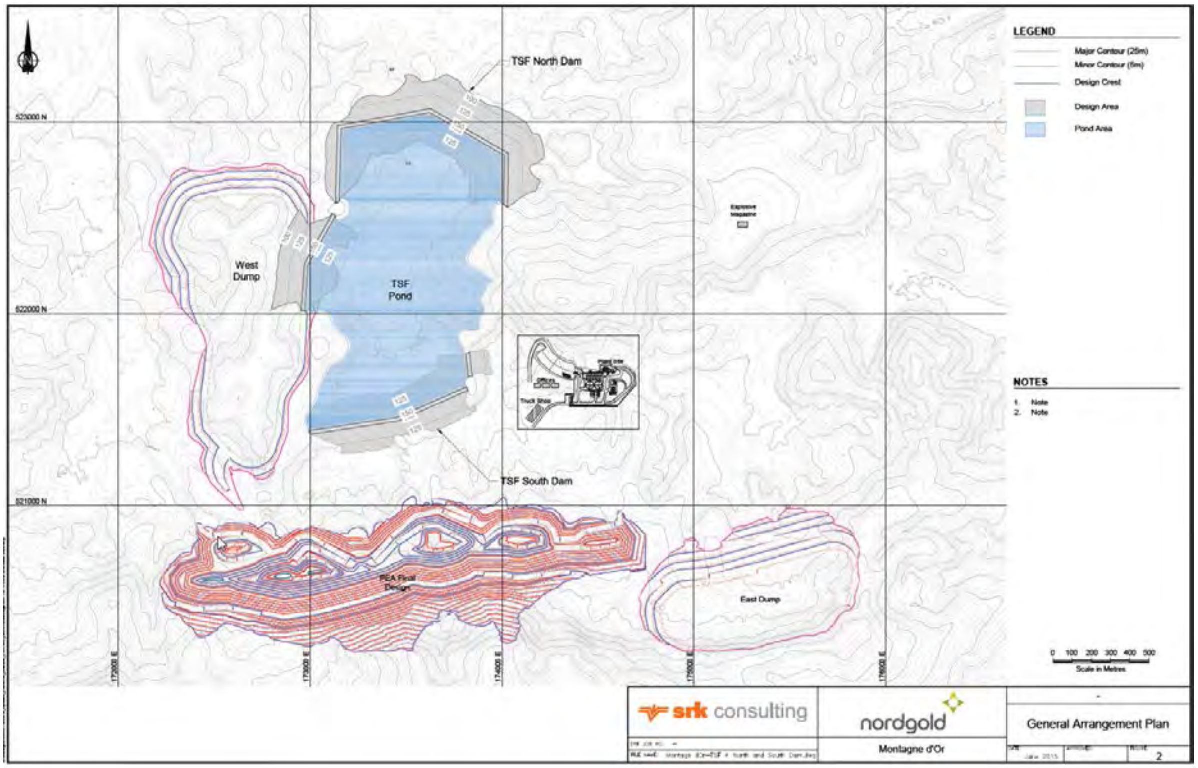


Figure 1.9.1: Near Mine Site Infrastructure

1.10 Environmental Studies and Permitting

Environmental and social baseline data collection was initiated by WSP Canada Inc. in 2014. Current findings to date are presented in the Preliminary Environmental Report (WSP, 2015), which also provides an early indication of the positive and potentially negative impacts associated with the planned operation. The intent this PEA is to provide direction for the continuing environmental assessment process, and guide the environmental authorities with the information required to determine the range of information and degree of detail needed in the formal impact assessment.

The operation is currently permitted for all of the activities associated with the exploration program from which this PEA has been prepared. Additional permitting will be necessary in order to move into the exploitation phase of the Project. Initiation of this permitting will likely occur during the preparation of a feasibility study (FS), and will include a detailed Environmental and Social Impact Assessment (ESIA) based on the FS design of the operation. Data from the Preliminary Environmental Report (WSP, 2015) will be utilized for the ESIA.

In addition to the land restrictions presented by the SDOM, the Project is located adjacent to a nature reserve, the Réserve Biologique Domaniale Lucifer Dékou-Dékou, managed by the ONF. Its Management Plan from the ONF is yet to be ratified, so there is little guidance or decisions regarding the use of land and allowable activities within the reserve. The boundaries of this reserve overlap four of the eight Paul Isnard mineral concessions however only one of these concessions is important to the project. Since these concessions already exist, and there has been continued exploration and mining activity in the area for over 100 years, the ONF has agreed to create several zones within the reserve boundaries where mining is permitted. The Montagne d'Or deposit itself is within a zone where open pit mining is permitted and the outer limit of the resource pit shell is located approximately 240 m from the reserve boundary.

Based on the preliminary geochemical characterization program initiated for mineralized material, waste rock and tailings, the potential for acid generation and leaching of metals remains a concern at this stage of the Project, and will need to be considered during design and development of the mine with respect to appropriate waste rock and tailings management.

By law, reclamation of the mine site following closure is required. The operator is required to restore the site to a state that is, at a minimum, similar to that described in the Baseline Report. Given the current lack of mine design information, the costs associated with closure of Montagne d'Or have been estimated at approximately US$25 million based on similar nature and extent of the operations to projects previously evaluated by SRK. This number will be refined using actual mine designs and country-specific costing rates during development of the FS.

1.11 Capital and Operating Costs

LoM total operating costs are tabulated in Table 1.11.1 and total US$31.83/t mill feed.

Table 1.11.1: Total Operating Cost Summary

Description	US$/t Mill Feed	LoM (US$000's)
Mining	11.38	635,356
Processing	14.45	811,997
Tailings	0.47	26,309
Support	5.42	302,724
Total	**$31.83**	**$1,776,387**

Source: SRK, 2015

Total capital costs are tabulated in Table 1.11.2 and include US$366 million of initial capital cost and US$216 million of sustaining capital cost. Reclamation/Closure costs are estimated at US$25 million.

Table 1.11.2: Total Capital Cost Summary (US$000's)

Description	Initial	Sustaining	Post Closure	LoM
Pre-Stripping	27,027	152,692		179,719
Open Pit Mining	53,513	33,143		86,656
Processing	136,741	0		136,741
Tailings	19,410	30,267		49,677
Infrastructure	70,500	0		70,500
Owner's Cost	14,875	0		14,875
Reclamation/Closure/Equipment Salvage	0	0	25,000	25,000
Subtotal	**$322,066**	**$216,102**	**$25,000**	**$563,168**
Contingency (14% of Initial capital cost)	44,360	0	0	44,360
Total Capital	**$366,425**	**$216,102**	**$25,000**	**$607,527**

Source: SRK, 2015

1.11.1 Mining Operating Costs

SRK estimated the mine operating costs on the prepared production schedule and selected mine equipment fleet. Table 1.11.1.1 presents the summary of the mine operating costs which are estimated at US$11.38/t processed (US$1.88/t mined). The major contributors to operating costs are hauling and labor.

Table 1.11.1.1: Mine Operating Cost Summary

Description	US$/t Mill	LoM (US$000's)
Drilling	0.85	47,547
Blasting	1.79	100,084
Loading	1.31	72,975
Hauling	5.67	316,320
Roads & Dumps	1.37	76,662
Labor	3.61	201,487
Re-Handling	0.00	0
Subtotal Open Pit	**$14.61**	**$815,076**
Cost Capitalized to Pre-stripping	(3.23)	**(179,719)**
Total Open Pit	**$11.38**	**$635,356**

Source: SRK, 2015

1.11.2 Process Operating Costs

Process operating costs are summarized in Table 1.11.2.1 and are estimated at US$14.55/t processed. Operating costs have been estimated by major categories (labor, power, consumables, etc.) and are based on a throughput capacity of 12,500 t/d. The major contributors to operating cost are power and reagents.

Table 1.11.2.1: Summary of Process Plant Operating Costs

Area	US$/t
Labor	1.50
Comminution Consumables	1.70
Reagents	4.50
Power	6.00
Maintenance Supplies	0.50
Other	0.35
Total Processing Costs	**$14.55**

Source: SRK, 2015

1.11.3 Tailings Operating Costs

Tailings operating costs are summarized in Table 1.11.3.1 and are estimated at US$0.47/t processed.

Table 1.11.3.1: Summary of Tailings Operating Costs

Description	US$/t Mill Feed	LoM (US$000's)
Total Tailings	**$0.47**	**$26,309**

Source: SRK, 2015

1.11.4 Support Operating Costs

Support costs were estimated by SRK at US$5.42/t mill feed as shown in Table 1.11.4.1. This unit rate is a placeholder based on similar analogous projects and not a build up from first principles.

Description	US$/t Mill Feed	LoM (US$000's)
General Facilities	4.30	240,224
Site G&A	1.12	62,500
Total Support	**$5.42**	**$302,724**

Table 1.11.4.1: Site G&A Operating Costs

Source: SRK, 2015

1.11.5 Mining Capital Costs

The estimated cost of mine equipment and timing of purchases are shown in Table 1.11.5.1. Mine capital equipment costs were obtained from recent cost models and handbooks.

Table 1.11.5.1: Open Pit Mining Capital Costs (US$000's)

Description	Initial	Sustaining	LoM
Drilling	2,664	3,374	6,038
Loading	13,521	6,386	19,907
Hauling	29,092	17,026	46,118
Roads & Dumps	8,236	6,357	14,593
Total Open Pit Mining	**$53,513**	**$33,143**	**$86,656**

Source: SRK, 2015

1.11.6 Process Capital Costs

The capital cost for the 12,500 t/d process plant is summarized in Table 1.11.6.1 and is estimated at US$136.7 million and is considered at a conceptual level with a +/-50% level of accuracy. The capital cost estimate is based on Infomine's CostMine Model for a CIP processing plant, and includes the following adjustments:

- Capital cost has been escalated to the 12,500 t/d design using the industry accepted Cost-Capacity relationship;
  - o $Cost_{p2} = Cost_{p1} X (Capacity_{p2}/Capacity_{p1})^{0.65}$;
- Tailings pond capital cost has been excluded (treated as a separate cost area);
- Working capital has been excluded (included in the technical economic model (TEM));
- Process plant capital cost has been increased by 30% based on SRK's experience with the CostMine models; and
- Maintenance sustaining capital is calculated as an operating expense.

Table 1.11.6.1: Process Plant Capital Cost Estimate (US$000s)

By Category	US$000's
Equipment	48,269
Installation Labor	30,630
Concrete	3,965
Piping	12,717
Structural Steel	4,376
Instrumentation	3,008
Insulation	1,504
Electrical	6,153
Coatings & Sealants	547
Mill Building	8,204
Engineering/Management	17,366
Total (by Category)	**$136,741**
By Area	**US$000's**
Comminution	42,350
CIL Leaching	30,240
Solid-Liquid Separation	8,986
General	10,480
Engineering/Management	13,129
Total (by Area)	**$105,185**
Capital Cost Adjustment (30%)	30%
Total Process Capital Cost	**$136,741**

- Working Capital Excluded;
- TSF Starter Dam Excluded;
- CIP capacity escalation factor = .65; and
- Info Mine Model Capital Cost Adjustment Factor = 30%.

Source: SRK, 2015

1.11.7 Infrastructure Capital Costs

The capital cost for infrastructure, not related to mining and processing, is estimated at US$84.6 million, which includes a 20% contingency as shown in Table 1.11.7.1. No sustaining capital was estimated.

Table 1.11.7.1: Infrastructure Capital Cost Estimate (US$000's)

Description	Initial
HFO/Palm Oil Power Generation (28 MW Nominal)	33,000
Water Treatment Plant	12,500
All Other Infrastructure	25,000
Subtotal	**$70,500**
20% Contingency	14,100
Total Infrastructure	**$84,600**

Source: SRK 2015

1.11.8 Other Capital Costs

The capital cost for owners' cost and closure/reclamation is US$39.9 million as shown in Table 1.11.8.1. Capital costs associated with social programs have not been estimated for the Project. No sustaining capital was applicable for owners cost or closure cost at this time.

Table 1.11.8.1: Owners and Closure Capital Cost Estimate (US$000s)

Description	Initial	Post Closure	LoM
Owner's Cost	14,875		14,875
Reclamation/Closure/Equipment Salvage		25,000	25,000
Subtotal	**$14,875**	**$25,000**	**$39,875**

Source: SRK, 2015

1.12 Economic Analysis

Project economic results and estimated cash costs are summarized in Table 1.12.1, which shows an after-tax NPV 8% of US$324 million and an IRR of 23.0% as currently designed with a All-In Sustaining Cost (AISC) of US$711/oz.

Table 1.12.1: LoM After-Tax Indicative Economic Results (US$000's)

Description	Value
Market Prices	
Gold (US$/oz)	$1,200
Revenue	
Payable Gold (koz)	3,054
Total Revenue	**$3,664,612**
Operating Costs	
Mining	(635,356)
Processing	(811,997)
Tailings	(26,309)
General Facilities	(240,224)
Site G&A	(62,500)
Selling/Refining	(3,069)
Royalties	(176,082)
Total Operating Costs	**($1,955,538)**
Operating Margin (EBITDA)	**$1,709,074**
Taxes	
Income Tax	(345,397)
Total Taxes	**($345,397)**
Working Capital	(0)
Operating Cash Flow	**$1,363,677**
Capital	
Initial Capital	(366,425)
Sustaining Capital	(216,102)
Reclamation/Salvage Capital	(25,000)
Total Capital	**($607,527)**
Metrics	
Free Cash Flow	$756,150
NPV @: 8%	$324,430
IRR	23.0%
Undiscounted Payback from Start of Comm. Prod. (Years)	3.6
AISC ($/oz)	$711

Source: SRK, 2015

The Project is most sensitive to changes in gold price where a 20% decrease in price would drive the Project to breakeven NPV 8%. However, on the upside, a 5% increase in gold price or 10% decrease in either operating cost or capital cost would increase the IRR over 25%, which is a common mining investor metric. In addition, Table 1.12.2 shows price sensitivity at a series of discrete price points.

Table 1.12.2: Sensitivity Analysis at Various Gold Price Points

Gold Price (US$/oz)	NPV@8% (US$ millions)	IRR (%)
947	$0 (Breakeven)	8.0
1,000	68,495	11.6
1,100	196,471	17.6
1,200	324,430	23.0
1,300	452,388	28.1
1,400	580,347	32.8

Source: SRK, 2015

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.13 Conclusions and Recommendations

1.13.1 Geology and Resources

- Columbus has completed an industry standard exploration drilling program over an area of approximately 1 1/4 km^2;
- The results of the drilling have supported an industry standard resource estimation;
- Whittle™ pit shell optimizations host an Indicated Mineral Resource of 83 Mt at an average Au grade of 1.455 g/t containing 3.9 Moz of gold and an additional Inferred Mineral Resource of 22 Mt at an average Au grade of 1.550 g/t containing 1.1 Moz of gold;
- A multitask exploration drilling program is proposed. The program will target infill drilling in the areas of the proposed starter pit, infill drilling in the saprolite material and condemnation drilling in the potential areas of infrastructure;
- The infill drilling program would be on a 25 m x 50 m grid spacing in the proposed area of the current resource starter pit. The drillholes are proposed to range from 35 to 320 m in length. Many of the holes would be drilled by RC to the maximum depth achievable and then taken to final depth with core. A total of 17,750 m in 123 drillholes would be required; and
- The condemnation drilling program will cover three areas of infrastructure including, the proposed plant site, the proposed waste rock site and the proposed tailings facility. The condemnation drilling would be on a 55 m grid pattern and would consist of 75 m long inclined holes at -55° to the north or north east. A total of 4,900 m in 65 drillholes would be required.

1.13.2 Open Pit Geotechnical Program

The following is a partial list of geotechnical data and information gaps that should be addressed as a part of advancing the project to a feasibility-level study:

- Rock strength testing. A rock strength testing program should be; conducted;
- Saprolite characterization and testing;
- Geotechnical specific drillholes that target pit walls at approximately 90° and provide an unbiased orientation to better understand discontinuity sets in the rocks;

- Geotechnical model. All available geotechnical data should be domained by geology and fault blocks. The domains should be analyzed and geotechnical parameter distributions should be incorporated into a rock mass model;
- Bench, inter-ramp and overall slope stability analysis should be completed for the open pit design, analyzing each wall orientation and rock mass domain to optimize pit slope angles
- The stability analysis of the pit should incorporate geohydrology and groundwater surface information; and
- All available data and analysis should be documented in a technical report.

1.13.3 Mineral Processing and Metallurgical Testing

- The metallurgical test program was conducted on two master composites formulated from available whole core intervals representing the UFZ and the LFZ, as well as selected variability composites;
- Three process options, including whole-ore cyanidation, a combination of gravity concentration followed by cyanidation of gravity tailing, and gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate, were investigated on two master composites, and the preferred process option and optimal conditions were further verified on ten variability test composites; and
- Processing by gravity concentration followed by cyanidation of the gravity tailings yielded the highest overall gold recoveries and was selected at the preferred process option. Gold recovery is projected at approximately 95% with this process option.

1.13.4 Mining

- Due to the amount of pioneering work that will be conducted in a rainforest environment with considerable amounts of saprolite on the side of a hill, the ability for trucks to operate efficiently will be vital for successful execution of the mine plan. It is recommended that during the FS, the geomechanical properties of benign rocks (no sulfidation or NAG rocks) be tested for suitability as a road course for haul roads. In addition to the use of some waste rock from the pit, SRK recommends that a quarry site be searched for, either as part of the sterilization drill program or geological interpretation. If a quarry is not possible then a source of laterite that can be screened for fines would also be suitable;
- SRK recommends that a FOS analysis on the pit walls be conducted as soon as possible. This will help determine the groundwater and geomechanical properties to be collected that will assist in the generation of final pit wall angles for the FS; and
- SRK recommends that a NAG/PAG Acid Rock Drainage (ARD) model be built for the classification of the waste rock types that require encapsulated disposal, or which can be used for other purposes. Metals based accounting should also be considered as part of this exercise as it is evident that there is acid neutralizing potential in some of the waste rocks.

1.13.5 Recovery Methods

- The selected process flowsheet will include gravity concentration followed by cyanidation of the gravity tailings to recover the contained gold and silver values, and will incorporate process unit operations that are standard to the industry, including: crushing, grinding, agitated cyanide leaching, gold and silver adsorption onto activated carbon, gold and silver desorption, electrowinning and refining;

- Process operating costs are estimated at US$14.55/t processed. Operating costs have been estimated by major categories (labor, power, consumables, etc.) and are based on a throughput capacity of 12,500 t/d. The major contributors to operating cost are power and reagents; and
- The capital cost for the 12,500 t/d process plant is estimated at US$136.7 million and is considered at a conceptual level with a +/-50% level of accuracy.

1.13.6 Tailings and Infrastructure

A site water balance should be conducted with the aim of the tailings storage facility (TSF) design to provide a net neutral water balance. This would prevent any discharge that would mean a water treatment plant would not be needed. SRK recommends that further work in this regard be continued in the FS.

The trade off between grid power generation and on site generation is preliminary. As the Project develops, the trade off between future energy supplies in-country versus the cost of fuel importation for site generation may vary from the assumptions made in this report. As such, SRK recommends both options continue to be evaluated during the FS and beyond.

The ground conditions of the TSF earthen embankment will require geotechnical investigation for stability purposes.

- Additional geochemical testing on a sample representative of the supernatant pond waters produced when liberated waters from the CIL tailings;
- Additional geochemical testing on samples representative of the waste rock being used for construction of the TSF dam;
- Final design-level subsurface site investigations in select areas including geotechnical laboratory testing, including a study to estimate available borrow material quantities;
- Final design-level study and design of the run-off collection channels and ponds;
- Final design of tailings distribution system and water reclaim system considering a potential economic trade-off study for different system options; and
- Conduct a site specific seismic hazard assessment.

1.13.7 Environmental

Given the results of the geochemical characterization to date, the program should be expanded to include additional samples of mineralized material and waste rock from around the deposit, as well as post-process tailings. Early indications are that additional active management of waste rock and tailings may be necessary in the hot and humid climate of French Guiana.

A site-wide, soil mercury contamination program should be considered to more accurately define the nature and extent of pre-mine contamination by illegal artisanal mining operations.

1.13.8 Projected Economics

- The Project estimates economic results using US$1,200/oz gold price with NPV 8% at US$324 million and 23.0% IRR. The Project, as currently designed with an Initial Capital cost of US$366 million for the 13 year mine life year mine life at a total cash cost of US$711/oz;

- For the first 11 years when stockpiles are not fed to the mill, the annual recovered gold ounce production is approximately 265 koz/yr;
- The Project NPV 8% changes by approximately US$1.1 million per dollar change in gold price; and
- Mining taxation assumptions should be investigated further due to current uncertainty in French tax code.

2 Introduction

2.1 Terms of Reference and Purpose of the Report

This report was prepared as a National Instrument 43-101 (NI 43-101) Technical Report, Preliminary Economic Assessment (PEA) for Nord Gold N.V. (Nordgold) with Columbus Gold Corporation (Columbus) by SRK Consulting (U.S.), Inc. (SRK) on the Project located in French Guiana. Columbus is the Project owner/operator and is currently exploring the deposit under an option agreement with Nordgold. The details of the option area agreement are discussed in Section 4.2. Nordgold has contracted with SRK for this technical study. The Project is operated under a local enterprise named Société de Travaux Publiques et de Mines Aurifères de Guyane (SOTRAPMAG) which is a 100% owned subsidiary of Columbus Gold.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Columbus subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Columbus and Nordgold to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Columbus. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

This report provides Mineral Resource estimates, and a classification of resources prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014.

2.2 Qualifications of Consultants (SRK)

The Consultants preparing this technical report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in Nordgold or Columbus. The Consultants are not insiders, associates, or affiliates of Nordgold or Columbus. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Nordgold or Columbus and the

Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions. The QP's are responsible for specific sections as follows:

- Bart Stryhas, Principal Resource Geologist, is the QP responsible for background, geology and resource estimation Sections 2 to 12, 14 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Bret Swanson, Practice Leader/Principal Consultant (Mining Engineer), is the QP responsible for mine design and mine planning Sections 15, 16, 18, 23, 24, 27 and 28 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Eric Olin, Principal Consultant (Metallurgy), is the QP responsible for Mineral Processing, Metallurgy and Recovery Sections 13 (except 13.10), 17, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Grant A. Malensek, Principal Consultant (Mineral Economics) is the QP responsible for Economics Sections 13.10, 19, 21 and 22, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Mark A. Willow, MSc, CEM, SME-RM, Principal Environmental Scientist/Practice Leader, is the QP responsible for environmental studies, permitting and social or community impact Section 20 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

2.3 Details of Inspection

Bart Stryhas, Bret Swanson and Mark Willow visited the Project site for three days on April 1-3, 2014. Over the three day visit, the group toured the general areas of mineralization, historic mining, drilling sites, reviewed existing infrastructure, observed the Columbus drill core and reviewed Project data files with Columbus' and Nordgold's technical staff.

Table 2.3.1: Site Visit Participants

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Bart Stryhas	SRK	Geology/Resources	April 1-3, 2014	Drill Core/ Field Geology
Bret Swanson	SRK	Mining	April 1-3, 2014	Project area
Mark Willow	SRK	Environmental	April 1-3, 2014	Project area

Source: SRK, 2015

2.4 Sources of Information

The sources of information include data and reports supplied by Columbus personnel as well as documents cited throughout the report and referenced in Section 27. The electronic database was compiled and transmitted by Columbus.

2.5 Effective Date

The effective date of this report is June 22, 2015.

2.6 Units of Measure

The metric system has been used throughout this report. Tonnes are dry metric of 1,000 kg, or 2,204.6 lb. All currency is in U.S. dollars (US$) unless otherwise stated. The Euro-US dollar conversion used in this report is based on an exchange rate of US$1.06:€1.00.

3 Reliance on Other Experts

The Consultant's opinion contained herein is based on information provided to the Consultants by Columbus throughout the course of the investigations. SRK has relied upon the work of other consultants in the Project areas in support of this Technical Report.

SRK has relied on Columbus's legal representation to describe the:

- Geopolitical;
- Mineral Rights;
- Nature and Extent of Ownership, and
- Royalties, Agreements and Encumbrances.

The majority of the text included in Sections 4 through 11 is taken from previous technical reports, and SRK has referenced these citations where used. Portions of these sections have subsequently been modified by Columbus staff and reviewed by SRK for compliancy with NI 43-101.The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

4 Property Description and Location

Montagne d'Or is located along the northern flank of the Dékou-Dékou range. Montagne d'Or is part of the larger Paul Isnard Project (Project). The Project consists of eight mining concessions and two pending exploration permit applications covering a total area of 190 km^2, located in the commune of Saint-Laurent-du-Maroni, NW French Guiana. The Project also includes historic artisanal mining operations, exploration roads and drill pads, a core logging/storage facility and Camp Citron. The camp hosts a main cook shack/office building and approximately six bunkhouse/shower buildings.

4.1 Property Location

The Project area and mining concessions are located in the northwestern portion of French Guiana, South America (Figure 4.1.1). The Project area extends from longitude 53° 53' 52'' W (UTM 178,475) to 54° 03' 09'' W (UTM 161,360), and latitude 4° 40' 59'' N (UTM 518,322) to 4° 51' 03'' N (UTM 536,922). Camp Citron, the base camp for the Project, is located approximately 4 km northwest of the deposit.

Source: Columbus, 2015

Figure 4.1.1: Paul Isnard Project General Location Map

4.2 Mineral Titles

4.2.1 Geopolitical

French Guiana is both a Region and a Department of France and is subject to French laws, with certain modifications and differences that are applicable to the Départements d'Outre Mer (overseas departments). The Region is governed by the President of the Region. The Department is governed by the President of the Department. Both are elected by the people of French Guiana. There is an election scheduled for December 2015 that will elect only one President to govern the merged Region and Department. The local administration is governed under the direction of the Prefect, who is appointed by the President of France and is the representative of the French government. In overseas departments, the Prefect has more extensive powers than their counterpart in mainland France. Mining is a national matter presided over by the Prefect.

SDOM Mining Legislation

The President of the French Republic, Mr. Sarkozy (at the time of legislation), committed himself to a new comprehensive mining legislation in French Guiana following his rejection, in February 2008, of IAMGOLD's development application for the Camp Caiman gold deposit. The mining project demonstrated the difficulties and contradictions related to the compatibility of industrial development and the protection of the environment in the Department.

The new mining legislation, referred to as the *Schéma Départemental D'Orientation Minière de la Guyane* (SDOM), was drafted by representatives of the national government of France in the Prefecture of French Guiana following broad consultation with regional communities, the economic players concerned, environmental protection organizations, trade unions, the State and local and regional bodies competent in the fields of natural and human environment, biodiversity and geology. The final SDOM legislation was approved by decree (*décret n^{o} 2011-2106*) on December 30, 2011, by the *Conseil d'État* (State Council), the highest administrative court in France, and went into effect on January 1, 2012.

The legislation was created with the dual objectives of encouraging economic development of the mining industry in French Guiana while protecting its environment and provides incentive, including security of land tenure and clear guidelines to mining development and environmental conditions and restrictions, to serious and environmentally responsible mining companies while inhibiting environmentally damaging illegal mining activities.

Under the SDOM legislation, the territory of French Guiana is divided into four land use classifications, defined as Zones 0, 1, 2 and 3 (the SDOM Zones), that clearly outline areas where the possibility of prospecting and mining are defined in accordance with Article L.621-1 of the *code minier* (Mining Code). The classification takes into consideration the necessity to protect sensitive natural environments, landscapes, sites and populations, a balanced management of the land and the natural resources, economic interests, and sustainable development of the mining resources, within the limits of current knowledge of the biodiversity and the mineral wealth. The areas where mining activity are permitted represents 55% of the territory:

- Zone 0: Banned for prospecting and mining.
- Zone 1: Open to airborne surveys, underground mining authorized subject to conditions.

- Zone 2: Open to prospecting, underground and open pit mining authorized subject to conditions.
- Zone 3: Open to prospecting and underground and open pit mining.

The Montagne d'Or gold deposit is located within an area classified as a favorable zonation (Zone 2), where all prospecting and mining activity is authorized, although subject to conditions as it lies in proximity to the Lucifer Dékou-Dékou biological reserves (RBI LDD).

Conditions to mining in Zone 2, which in actual fact would be applicable to large scale commercial mining operations anywhere in French Guiana include:

- Demonstration of a viable mineral deposit;
- Adherence to a Charter of Good Practices approved by the State representatives;
- Completion of an Environmental Impact Study and Reclamation Plan; and
- Requirements in Zone 2 can include additional reclamation or environmental investigations as may be required for the public interest, on or off site.

Lucifer and Dékou-Dékou Biological Reserve

The initial Lucifer Dékou-Dékou domanial biological reserve (RBD LDD) was created in 1995 over an area covering 110,300 hectares.

Following the implementation of the SDOM legislation, an Order by the Ministry of *l'écologie, du dévelopement durable et de l'énergie* (EDDE) and the Ministry of *l'agriculture, de l'agroalimentaire et de la forêt* (METL), referred to as the '*Arrêté du 27 juillet 2012'*, was issued in July, 2012, to create and establish the boundaries of the RBI LDD. The biological reserve covers 64,373 hectares and is administered by the ONF.

The principal objectives of the biological reserve is to permit the evolution of the natural forest ecosystem, the preservation of biological diversity and improving scientific knowledge on the Lucifer and the Dékou-Dékou massifs. To attain these goals human activity within the biological reserve are regulated and logging, prospecting and mining are prohibited.

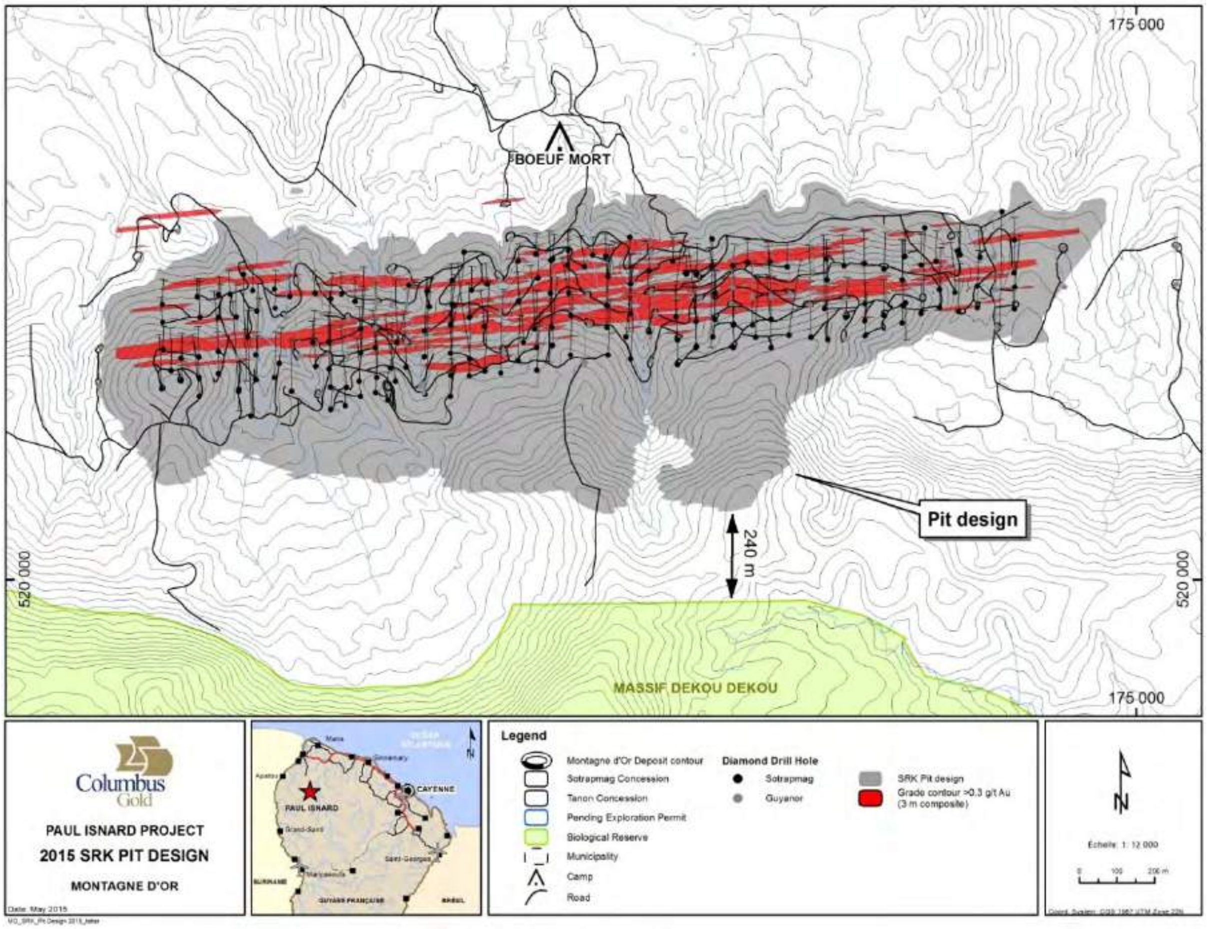
The RBI LDD is separated into two domains located immediately north and east and south of the Paul Isnard concessions, referred to as Lucifer and Dékou-Dékou, respectively.

To the south of the Montagne d'Or Mineral Resource, the boundary of the Dékou-Dékou portion of the biological reserve is defined from west to east by:

- The 420 m elevation line over a distance of 5.5 km;
- A 0.8 km straight line oriented 107^{o} azimuth starting at the 420 m elevation extending to the 505 m elevation and then rejoining the 420 m elevation; and
- Extending southeast along the Apollon creek bed over a distance of 2.8 km.

The location of the Dékou-Dékou biological reserve with respect to the potential resource pit outline is shown in Figure 4.2.1.1. There is currently a 240 m set-back between the reserve boundary and the potential pit outline.

The southern portion of the concession C02/46 that falls within the RBI LLD are open to airborne surveys and underground mining (Zone 1).



Source: Columbus, 2015

Figure 4.2.1.1: Location of the Potential Resource Pit Outline and Biological Reserve

4.2.2 Mineral Rights and Properties

Mineral exploration and mining are subject to the provisions of the *code minier*, which specifies that the State can grant to an operator a right to prospect or exploit the Mineral Resources over a specified area and period.

Special regulations have been established for the Department of French Guiana to take into account certain distinctions specific to this territory (law no98-297 of April 21, 1998). In addition to the *code minier*, that include Exclusive Research Permits (PER) for prospecting and Concessions for mining, the regulations concerning French Guiana provide for Mining Research Authorizations (ARM), in areas managed by the ONF, Exploitation Authorizations (AEX) and Exploitation Permits (PEX).

Mineral rights and mining are administered by the *Direction de l'environnement*, *de l'aménagement et du logement* (DEAL) under the authority of the Prefect. Their locations are reported in UTM, World Geodetic System (WGS) 84, Zone 22.

Exclusive Research Permit (PER)

In general, the PER is the initial permit application to conduct prospecting.

- Maximum area: No restriction. The area has to fit reasonably with the exploration objectives and the geological context.
- Dimensions & Form: No restrictions, as long as protected areas are not included within the area requested.
- Maximum period: 15 years. Initial application is for 5 years, twice renewable for up to 5 years. Surface area can be reduced by 50% in each renewal application. Following the extensions it is required to apply for a Concession or Exploitation Permit.
- Restriction: The initial application is open to competitor bidding if it covers an area greater than 50 km^2.
- Requirements: Financial commitments are based on the exploration program and expenditures proposed in the mining title application, which need to be in accordance with the surface area of the mining title. Conditions of renewal are based on the completion of the financial commitments in the corresponding period.

Exploitation Permits

Mining in French Guiana is permitted under the following permits:

- Concession;
- Exploitation Permit (PEX); and
- Exploitation Authorization (AEX).

PEX and AEX are exclusive to the *départements d'Outre-Mer*, such as French Guiana.

Concession

- Maximum area: No restriction.
- Dimensions & Form: No restrictions.
- Period: 50 years. Renewable by 25-year tranches if the mining operations are active at time of renewal. All the concessions, in French Guiana, will expire by December 31st, 2018. On the concessions, there are no financial commitments. However, for a concession to be able to be renewed, its owner has to prove a gold production (from itself or from any company legally exploiting gold on the concession) on the concession before December 31st, 2018.
- Restriction: Open to competitor bidding unless it arises from a PEX or PER.

Exploitation Permit (PEX)

- Specific disposition: Medium-scale alluvial and small-scale vein-type mining.
- Maximum area: No restriction.
- Dimensions & Form: No restrictions.
- Maximum Period: 15 years. Initial application is for 5 years, twice renewable for up to 5 years.
- Restriction: The initial application is open to competitor bidding unless it arises from a PER or if the total surface area is less of equal to 50 km^2.

Exploitation Authorization (AEX)

- Specific disposition: Small-scale artisanal mining, mainly for alluvial exploitations, sometimes for primary gold in saprolite.
- Maximum area: 1 km^2.
- Dimensions & Form: 1 km x 1 km or 0.5 km x 2 km.
- Maximum Period: 8 years. Initial application is for 4 years, once renewable for up to 4 years.
- Restrictions: Maximum of 3 AEX by *département d'Outre-Mer* in a same 4-year period. An AEX can be issued over an area covered by a PER, Concession or PEX with consent of the holder of these titles and as long as they are active. The holder of the PER, Concession or PEX loses all mineral rights over the area covered by the AEX.

The Project is composed of eight mining concessions which cover an area of approximately 135 km^2 (13,500 ha). The concessions are listed in Table 4.2.2.1 and shown in Figure 4.2.2.1.

Table 4.2.2.1: Land Tenure of the Paul Isnard Project

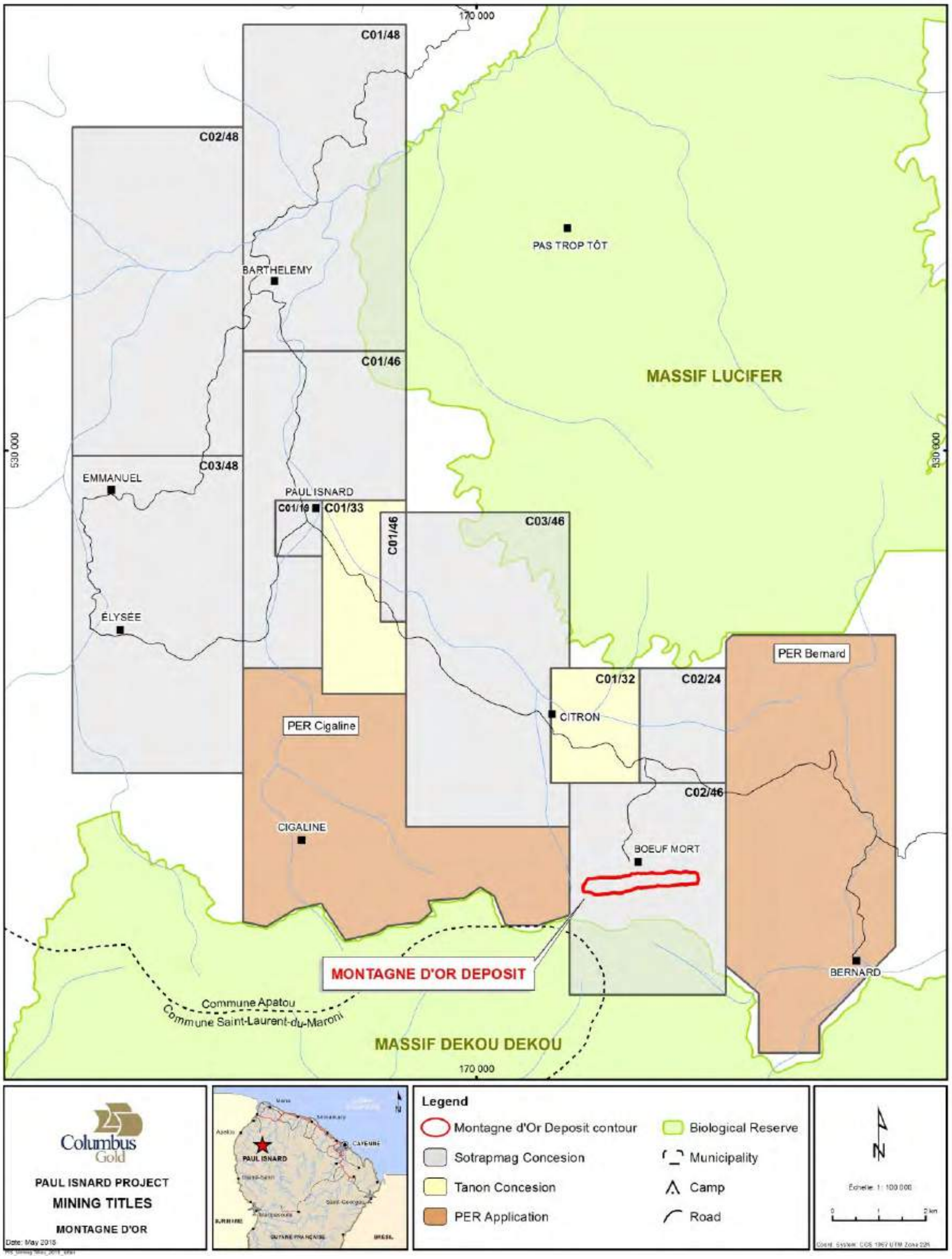
# Mining Title	Type	Surface km^2	Transfer to SOTRAPMAG	Expiry Date
C01/19	Concession	1.200	Decree : 12/27/1995 (JO : 12/29/1995)	12/31/2018
C02/24	Concession	4.471	Decree : 12/27/1995 (JO : 12/29/1995)	12/31/2018
C01/46	Concession	17.272	Decree : 12/27/1995 (JO : 12/29/1995)	12/31/2018
C02/46	Concession	15.075	Decree : 12/27/1995 (JO : 12/29/1995)	12/31/2018
C03/46	Concession	22.470	Decree : 12/27/1995 (JO : 12/29/1995)	12/31/2018
C01/48	Concession	24.500	Decree : 12/27/1995 (JO : 12/29/1995)	12/31/2018
C02/48	Concession	25.375	Decree : 12/27/1995 (JO : 12/29/1995)	12/31/2018
C03/48	Concession	24.469	Decree : 12/27/1995 (JO : 12/29/1995)	12/31/2018
Total		**134.832**		

Source: Columbus, 2015

Office National des Forêts Rights

As most of the ground in French Guiana belongs to the French State and is covered by the equatorial rainforest, the ONF was designated to manage the private domain of the State. Therefore, any occupation of the ground, in forested areas, is submitted to an authorization by the ONF (camps, access roads, etc.). Subject to application, the ONF grants land use permits or "*Convention d'Occupation Temporaire du Domaine Privé de l'Etat pour activités minières*" (COTAM) to mining title holders. SOTRAPMAG holds a COTAM dated April 24, 2009, valid until December 31, 2018, for the use of the road from Apatou Crossing to Citron (60 km) and for the surface area of Citron camp and airstrip. The COTAM has annual fees based on the surface area of the deforested land, kilometers of roads, and surface occupied. As an example, for the Paul Isnard project, SOTRAPMAG pays annual fees to the ONF for the use of the road from Apatou Crossing to Citron (5,400 €), for the surface area of Citron camp and airstrip (3,700 €), as well as for the opening of new access roads and drill pads (variable, but about 800 € for 2014). A *COTAM* will be necessary, in the future, for mine infrastructures and wastes and tailings sites.

Access to the Paul Isnard mining concessions is guaranteed by the existence of the mining titles under the right of access to the Mineral Resource ("*accès à la ressource*").



Source: Columbus, 2015

Figure 4.2.2.1: Location of Columbus Concessions and PER Applications

4.2.3 Nature and Extent of Issuer’s Interest

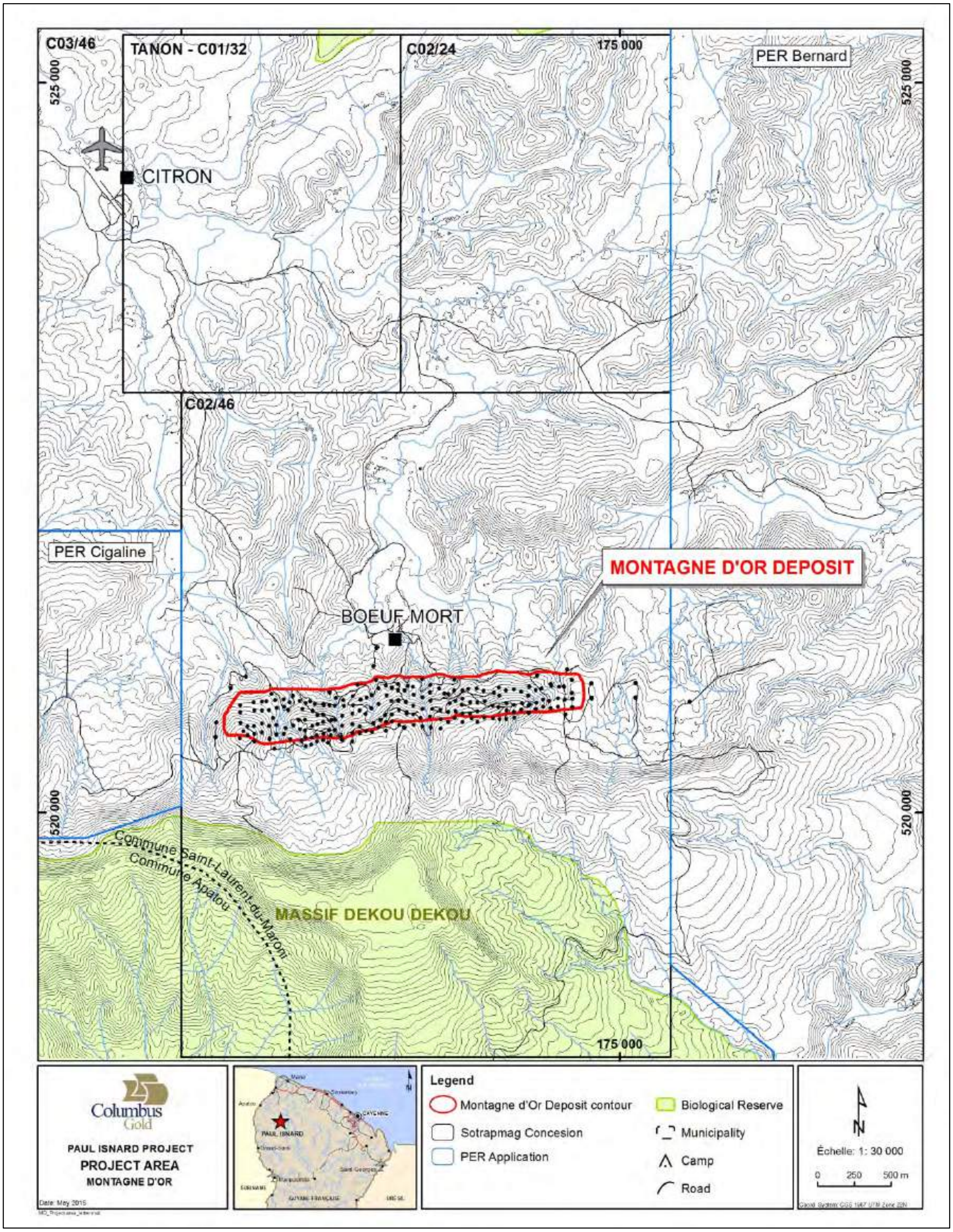
In November 2010, Columbus entered into an option agreement to acquire control of the Paul Isnard Project from Auplata SA. In January 2013, Columbus Gold completed the acquisition of a 100% interest in Paul Isnard.

On March 13, 2014, Columbus Gold and Nordgold signed the definitive option agreement pursuant to which Nordgold has the right to earn a 50.01% interest in the Paul Isnard Project and the pending PER applications (54.3 km²) within a three year option period terminating in March 2017.

4.2.4 Location of Mineralization and Facilities

The Montagne d’Or exploration area is located approximately halfway up the steep northern slope of the Dékou-Dékou Mountain within mineral concession C02/46 (215) shown in Figure 4.2.4.1. The mineralization and proposed mining and processing facilities, with the exception of the man camp, are within mineral concession C02/46.

The camp for the current exploration and the proposed mining operation could be located at Citron Camp. Citron Camp is within mineral concessions C01/46 held by SOTRAPMAG and C01/32 held by Tanon S.A. (Tanon). The access road crosses two Tanon held mineral concessions. The road crosses Tanon held mineral concessions C01/32 between the mineralized zone and Citron Camp and mineral concession C01/33 north of Citron Camp (Figure 4.2.2.1). Under the mining code, SOTRAPMAG has rights to any access roads leading to the Paul Isnard concessions.



Source: Columbus, 2015

Figure 4.2.4.1: Paul Isnard Project General Site Map

4.3 Royalties, Agreements and Encumbrances

The Paul Isnard Project is subject to a 1.0% NSR royalty payable to Sandstorm Gold Ltd.

There is also a NSR royalty of 1.8% on the first 2 Moz of gold produced and 0.9% on the next 3 Moz of gold produced on the Paul Isnard Project payable to Euro Ressources, an 86%-owned indirect subsidiary of IAMGOLD Corporation.

The royalty payable in French Guiana is for distribution to the local communes (towns), of €683.50 (US$724.51)/kg. In addition, there is a Communal tax of €132 (US$139.92)/kg and Departmental tax of €26.30 (US$27.88)/kg (2014). The Euro-US dollar conversion in this paragraph is based on an exchange rate of US$1.06: €1.00.

The Paul Isnard Project is also subject to reclamation of previous mining works, as described in Section 4.4.1, to a maximum expenditure of €350,000. The reclamation work is currently in progress and is expected to be completed in September 2015.

4.4 Environmental Liabilities and Permitting

4.4.1 Environmental Liabilities

The Project area is an intermittently active exploration property centered in dense tropical rain forest. Exploration activities require access road and drill pad construction, trenching, water management features, as well as construction of worker camps. Environmental liabilities resulting from previous and ongoing exploration activities are fairly limited due to the high precipitation and rapid natural rehabilitation that occurs in the rainforest. Holders of exploration permits are required by law to reclaim worked areas, control stormwater and potential sedimentation of downstream surface water resources, and are strictly prohibited from using mercury. These conditions are monitored closely by the government. The previous project owners, and by extension Columbus, negotiated an agreement with French regulatory authorities to dedicate up to €350,000 (US$396,000) to reclamation of exploration disturbances for which it is responsible.

While not the responsibility of Columbus Gold, illegal artisanal placer mining that occurs over much of the Project area has disturbed considerable land area, and continues to impact local surface water resources through increased sedimentation and mercury contamination.

4.4.2 Required Permits and Status

Discussion related to mining in French Guiana, the Mining and Environmental Codes, as well as the permits and authorizations necessary for mineral exploration and exploitation is provided in Section 20.3. In addition, some background into the anticipated mining code reforms is also provided.

4.5 Other Significant Factors and Risks

There are no known factors or risks that affect access, title or right or ability to perform work on the property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Section 5 has been excerpted from the Coffey 2014 Technical Report. Standardizations have been made to suite the format of this report. Changes made by SRK are indicated by the use of brackets [ ] or in sentences containing "SRK". Some spelling has been modified.

5.1 Accessibility

Montagne d'Or is located in the north-western portion of French Guiana, not far from the Maroni River that forms the border with Surinam. The property is accessible throughout the year by charter aircraft and by road that requires maintenance and upgrade. At Camp Citron, where the base camp is located at a distance of approximately 4 km from the Prospect area, there is a 500 m grass runway that can accommodate small aircraft. Alternatively, a helicopter charter service is available from Cayenne.

The flight from Cayenne to Paul Isnard takes approximately 55 minutes.

A forest road leads for a distance of approximately 125 km from Saint-Laurent-du-Maroni on the Maroni River to the Montagne d'Or prospect area. The first 65 km from Saint-Laurent-du-Maroni to Croisée d'Apatou is maintained by the State and supports all season travel. SOTRAPMAG has an exclusive right to use of the final 60 km of the road. This road section is currently being maintained by previous project owners to accommodate normal vehicle access for servicing the site.

Several roads that crisscross the mining concessions provide reasonable access for larger pickup trucks. Four wheel ATVs are used where access is prohibitive to pickup trucks. Access from Cayenne to the Project area is possible either by small plane or by helicopter (Figure 5.1.1), and takes approximately 50 to 55 minutes flying time to Citron.

Source: Columbus, 2015

Figure 5.1.1: Picture of Helipad/airstrip at Camp Citron

5.2 Climate

The climate is equatorial, with daytime temperatures between 29°C and 33°C, decreasing to 19°C to 23°C at night. There are two wet seasons; the main period is typically from April to the end of August, and the lesser period lasts from mid-November to mid-March. Average annual rainfall is in excess of 2,000 mm with a minimum monthly rainfall of 50 mm. Humidity is constantly high and typically ranges between 78% and 92%. The operating season is year-round.

5.3 Resources and Infrastructure

Skilled, semi-skilled and unskilled labor is readily available in Cayenne, with most professional and technical personnel being trained in Metropolitan France. Unskilled labor is also available in Saint-Laurent-du-Maroni. As French Guiana is a Department of France, French labor laws apply, resulting in relatively high salaries and restrictive employment contracts when compared to the neighboring countries of Surinam and Brazil.

Camp Citron infrastructures are 100% owned by SOTRAPMAG. A land use permit for the camp area and airstrip was obtained by Euro Ressources April 24, 2009. The permit is valid until December 31, 2018 on the expiry date of the concessions (ONF-Euro_Convention_2009-04-24).

Sufficiency of surface rights for mining operations, the availability and sources of power, water, mining personnel, potential tailings storage areas, potential waste disposal areas, heap leach pad areas, and potential processing plant sites have not yet been established for this exploration Project.

5.4 Physiography

Most of the region is covered by a thick canopy of primary and secondary tropical forest. The larger valleys have been extensively worked by alluvial miners in the past and are generally covered by thinner secondary forest or grassy-scrub and bamboo. Thick areas of bamboo are also present in many streams especially on the steeper slopes and in areas of old mine workings. The mean elevation is approximately 130 m ASL.

The general relief of the region is dominated by three geomorphological features:

- The east - west trending Massif Dékou-Dékou Range;
- The southwest - northeast trending duricrust plateau of Montagne Lucifer; and
- The northwest - southeast drainage system of the Roche River.

Montagne d'Or occupies the northern flank of the Dékou-Dékou Range, of which Montagne d'Or forms the northern flank.

There are numerous broad valleys, many of which have been exploited for their alluvial gold deposits. These are separated by areas of moderately rugged to more rounded hilly relief and often deeply incised valleys.

6 History

Section 6 has been excerpted from the Coffey 2014 Technical Report. Standardizations have been made to suite the format of this report. Changes made by SRK are indicated by the use of brackets [ ] or in sentences containing "SRK". Some spelling has been modified.

6.1 Prior Ownership and Exploration

The Paul Isnard concessions have been a regional center of alluvial and colluvial gold production since 1873 with some minor underground development in a few places. Beginning about 1890 bucket type dredging was undertaken and was replaced by dragline operations in 1949. Due to government permitting issues, little if any work was undertaken except by small illegal miners from 1950 to 1965 when placer mining recommenced and continued until approximately 1997.

The area was previously explored by the Bureau Minier Guyanais (BMG) and later the Bureau de Recherches Géologiques et Minières (BRGM), the French Geological Survey. This work confirmed the alluvial mining potential of the region and also located the primary Montagne d'Or prospect as a result of a regional geochemical program in 1976. This was not recognized as such until the data was reinterpreted in 1984. The BRGM undertook detailed surficial geochemical work and geological mapping.

The Paul Isnard Mine was started in 1956 by a company called SERMIG; gravel mining commenced in 1966 and continued for 20 years through an American company. Recovery was through an amalgamation plant and must have been poor. From 1986, a new owner (Pichet-Driss) obtained control, improved the process and operated the mine until 1993. SOTRAPMAG was involved in the gravel mining operation as a partner with the SGM, CERMI and Pichet-Driss.

In May 1993 Golden Star Resources Ltd endeavored to acquire title to the mine properties of the Paul Isnard Mine off SOTRAPMAG who was the owner of the mine and carried out a two-week evaluation of the operation. Total production from 1987 to 1993 was at this stage reported at 5,142 oz of gold and 354 oz of silver. This would roughly indicate a 7% average silver content of the gold doré.

Intensive exploration did not begin until 1994 when Guyanor Ressources S.A. ("Guyanor" approximately 70% owned by Golden Star Resources) had acquired the concessions and undertook regional scale remote sensing (LandSat, geophysics), geological examinations and geochemical surveys. Guyanor acquired the property in October 1994 through the 100% acquisition of the mining company SOTRAPMAG. Guyanor is registered in French Guiana with the right to explore deposits of gold, precious metals, base metals, and precious stones.

When Guyanor purchased SOTRAPMAG, it paid off an interest of Alcatel Alsthom Compagnie Générale d'Electricité (ALCATEL) in a primary deposit in the area to the BRGM while the company La Source Développement (LaSource) received an initial 25% participating interest. It is reported that LaSource did decide not to participate as a minority partner and that its interest was subsequently diluted.

From June 1996 until May 1998 exploration on the property was operated as a joint venture between SOTRAPMAG and Asarco Guyane Française with LaSource as a non-contributing partner. A PER was granted by Ministerial Decree (Official Bull. dated November 30, 1999) 100% to Guyanor (later named Euro Ressources) on November 26, 1999 for a period of three years from 1 December 1999 to 30 November 2002. Following the formation of the Joint Venture with Asarco and La Source, detailed geology, geochemistry and geophysics was completed along with 56 drillholes totaling for 10,916 m. In September 1999 the LaSource interest is reported as approximately 10% and that when it falls to below 10% it will convert to a 2.5% net proceeds royalty.

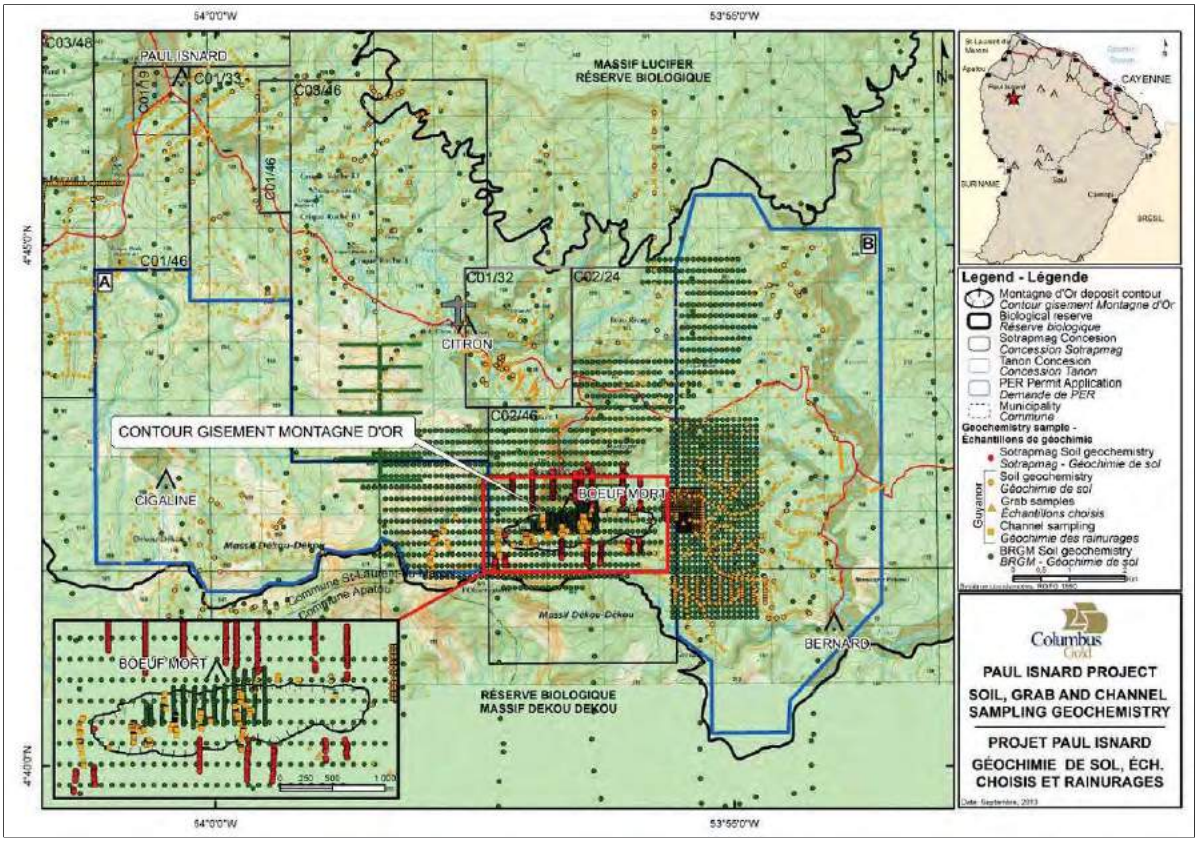
In 2001 a program of drilling was completed by Guyanor in conjunction with a JV agreement signed between Guyanor and Rio Tinto Mining and Exploration Ltd. Rio Tinto however concluded that the deposit did not have sufficient potential (more than 10 Moz) within saprolitic and near surface material to be mined by open pit methods followed by a cyanide recovery process.

Input to this study was mainly a re-interpretation of all available structural, geological and geophysical data and a study of older drill core obtained by Élysée (six diamond drillholes for 598.45 m) and Apollon (three diamond drillholes for 405.40 m), a regional geochemical soil program covering areas that were not previously covered (total of 1,058 soil samples) and a follow-up soil geochemistry and ground geophysics program (69 km) investigating the located anomalies. Selected anomalies were followed-up in 2001 with a limited diamond drilling program (Élysée six additional drillholes for 636.50 m, Paul Isnard three drillholes for 358.95 m, Citron three diamond drillholes for 343.50 m). One drillhole at Paul Isnard (Montagne d'Or) intersected a 7.0 m mineralized interval at 1.03 g/t Au. After completion of the program, Rio Tinto decided to withdraw from the JV.

Guyanor has carried out exploration activities in the areas at and around Montagne d'Or since 1994. Diamond drilling by Guyanor from 1996 (in JV with Asarco) to 1998 resulted in a total of 56 drillholes for 10,916 m. Guyanor also drilled 18 holes in 2001 in a JV with Rio Tinto and in 2007 Euro drilled one additional drillhole at Paul Isnard. Guyanor became Euro Ressources.

Until the property was acquired by Columbus in 2010, work done largely consisted of desktop evaluation of the resource potential and possible economic viability, and little additional exploration work was undertaken.

Since before 1900 up to around 1950, small scale alluvial mining has taken place in the area. This was followed by large scale alluvial mining from 1965 while the BRGM undertook geological mapping and regional geochemistry from 1930 to around 2000. Guyanor started work on the property in 1994. A regional overview of the various soil sampling, grab sampling and channel sampling programs is provided by the map in Figure 6.1.1.



Source: Coffey, 2014

Figure 6.1.1: Plan Map Overview of Historic Exploration Campaigns

6.2 Historical Mineral Resource Estimations

There have been five previous CIM compliant Mineral Resource estimations made of the Montagne d'Or prospect. These are summarized in Table 6.2.1. SRK notes the historical resources are not current Mineral Resources; they have been superseded by the current SRK Mineral Resource estimate discussed in Section 14 of this Technical Report. SRK has not done sufficient work to classify the historic estimates as current. The historical resources are provided here for information purposes only.

Table 6.2.1: Previous Resource Estimates for the Montagne d'Or Deposit

Year	Source	CIM Compliant	Resource Classification	Cut-off (g/t)	Tonnes (M)	Au (g/t)	Contained Au oz (M)
2004	RSG Global	Yes	Inferred	0.8	60.5	1.5	2.9
2008	SRK	Yes	Inferred	0.5	33.2	1.7	2.0
2011	SRK	Yes	Inferred	0.4	36.7	1.6	1.9
2012	Coffey Mining (Canada)	Yes	Inferred	0.3	115.2	1.44	5.3
2014	Coffey Mining (Australia)	Yes	Inferred	0.3	169.2	0.9	4.6

Source: SRK, 2015

7 Geological Setting and Mineralization

Section 7 has been partially excerpted from the Coffey 2014 Technical Report and Updated by Columbus current to this report. Standardizations have been made to suite the format of this report.

The Montagne d'Or deposit is composed of a bimodal felsic and mafic igneous units with lesser volcaniclastics towards the base of the sequence. The units strike east-northeast and dip steeply south. The eastern portion contains a preponderance of mafic volcanics relative to felsic volcanics. All geological units have been strongly deformed, as evidenced by a penetrative S1 foliation that locally transposes S0 and in places is mylonitic. The volcanic-plutonic package that hosts the deposit is tightly to isoclinally folded. The S1 foliation is constant throughout the section, striking on average 084° with an average 72°S dip. The intensity of deformation varies significantly over the distance of a few meters. The Project area is cross cut by post deformation diabase dikes that were apparently emplaced within northeast striking shears, faults or fractures that formed during a regional transcurrent tectonic event.

In general, the Montagne d'Or deposit consists of a number of tabular mineralized bodies within laminated, mainly felsic metavolcanic rocks. Mineralization has been encountered over a strike length of almost 2,500 m and to a vertical depth of at least 200 m. The mineralization is open at depth, along strike and internally between widely spaced holes.

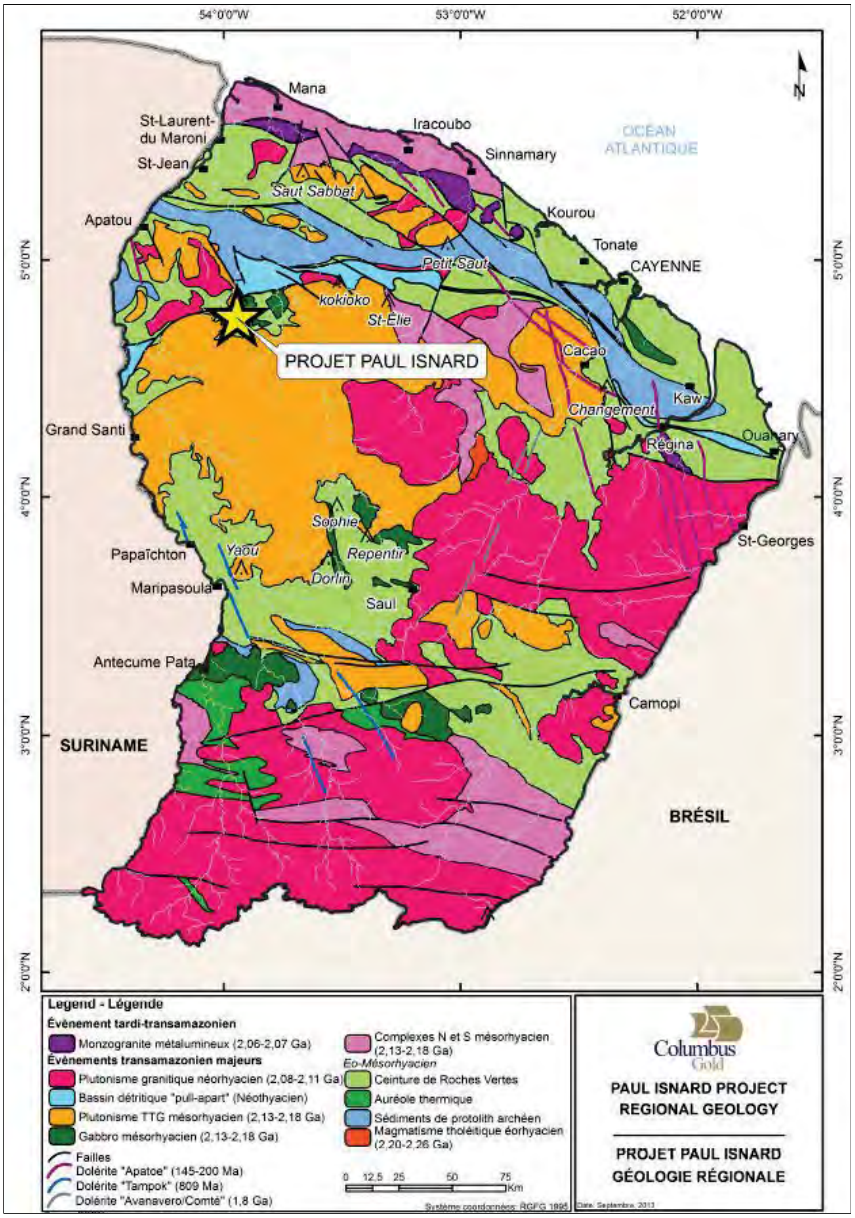
The mineralization appears as narrow elongated higher grade lenses within broader zones of low grade but anomalous mineralization (0.25 to 0.4 g/t Au). The main area of gold mineralization occurs in a series of generally east-northeast striking parallel zones with overall dimensions of 2,200 m x 400 m wide and to at least 200 m vertical depth. However, gold has been encountered outside the main zone of mineralization in the host rocks over a strike length of at least 3,500 m. Several distinct anomalous mineralized domains can be recognized that are separated by barren intercalated mafic and felsic rocks. Mineralization consists of semi-massive sulfide bands, as sulfidic stringers and as disseminated sulfides. Visible gold is present but rarely observed; preliminary mineralogical work suggests that it occurs along micro-fractures and on sulfide grain boundaries.

7.1 Regional Geology

The following is based mainly on work published by Milesi et al (2003) and Delors et al (2001), and on the most recent geological and structural interpretations carried out by a team from the Université du Québec à Montréal (UQÀM) and published in 2014 (Giraud et al, 2014). The latter studies also use and discuss historic and important geological interpretations by Vanderhaeghe et al (1998), and Franklin et al (2001). An earlier publication important for understanding the evolution of the geological interpretation of the French Guiana geology is the exploration report by Suter prepared for Guyanor in 1999.

The Paul Isnard concessions occur within the Guiana Shield, a large (approximately 900,000 km^2) segment of the Amazonian Craton of South America (Figure 7.1.1). The majority of the Guiana Shield formed during Proterozoic periods of intense magmatism, metamorphism and deformation that culminated in the Transamazonian tectono-thermal event of 2.1 to 1.9 Ga. The low-grade, volcanic-sedimentary greenstone sequences and affiliated granite intrusives that comprise the shield yield U-Pb age dates between 2.25 Ga and 2.08 Ga.

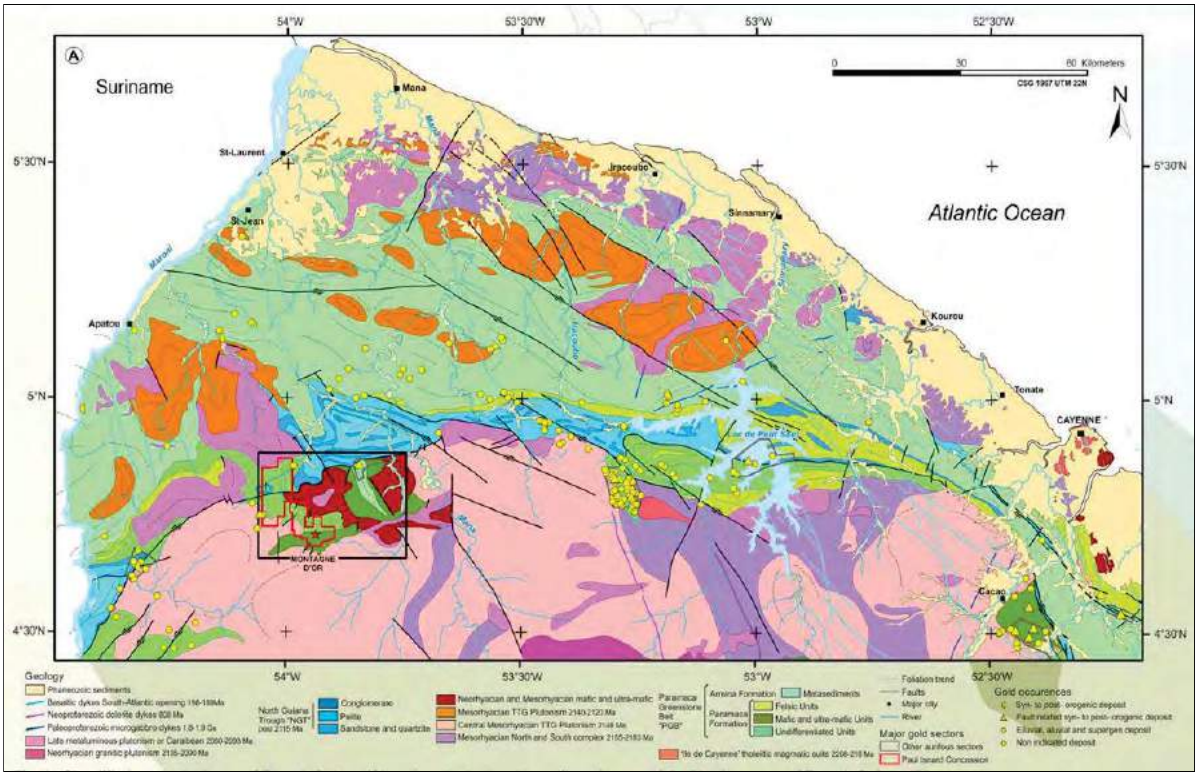
Major structural features include the Central Guiana Shear Zone (CGSZ) and the North Guiana Trough (Sillon Nord Guyanais, NGT). The CGSZ is a large-scale ductile shear zone, extending from French Guiana westerly through central Suriname and north-central Guyana. The NGT is interpreted to be a sinistral strike-slip "pull-apart basin" formed during one of the major tectonic stages of the Transamazonian Orogeny (Voicu et al, 2001).



Source: Coffey, 2014

Figure 7.1.1: Large Scale Geological Map of French Guiana

The greenstone belts of French Guiana are divided into two major groups. The northern group is associated with the NGT and includes the Lower Proterozoic Paramaca Greenstone Belt (PGB), a formation consisting of volcanic, volcaniclastic and sedimentary units. The PGB trends roughly from the west to the east through British Guiana, Dutch Guiana (Surinam) and French Guiana (Figure 7.1.2).



Source: Coffey, 2014

Figure 7.1.2: Large Scale Overview of the Geology of Northern French Guiana, showing the location of the Paul Isnard Project

Together with intrusive complexes of tonalite, trondhjemite and granodiorite, the PGB forms the Guiana Shield which was connected during the Paleozoic to the West African Shield (after Guiraud, Jébrak and Tremblay, UQÀM, April 2014). The PGB is interpreted as the remnant of a volcanic island-arc sequence that was tectonically deformed during the Transamazonian Orogeny, interpreted to be the result of plate convergence between the West African and the Guiana Shields.

This PGB occurs extensively across northern French Guiana, striking N110°E and hosting a number of gold deposits including Paul Isnard, Camp Caiman, St. Elie, Koolhoven and Rosebel in Surinam. The southern group is associated with the CGSZ and extends from Surinam through French Guiana. It includes sedimentary rocks of the Lower Orapu Formation and volcanic-sedimentary units of the Arima Formation (2.11 to 2.09 Ga), which unconformably overlie volcanic units of the PGB and the granite-gneiss complex of the Guianese Massif Central (2.3 to 2.2 Ga and 2.13 to 2.08 Ga). This southern group hosts gold mineralization at Benzdorp in Surinam, Yaou and Dorlin in French Guiana, and numerous other smaller workings. Most of the remainder of French Guiana is composed of the Lower Proterozoic granite-gneiss metamorphic complex of the Guianese Massif Central, and a central belt of Paramaca volcanic, volcaniclastic, and sedimentary lithologies.

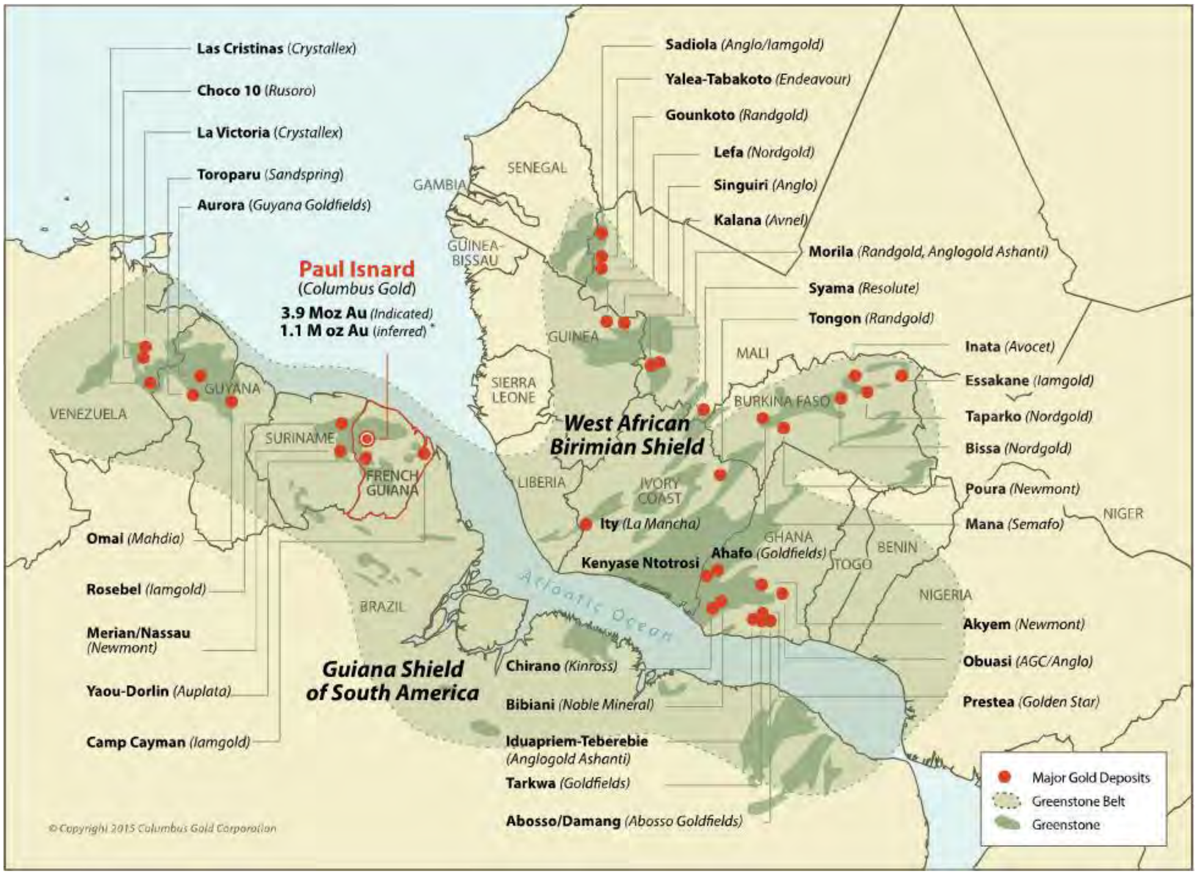
The northern and southern domains of the PGB are separated by the intrusions of tonalite, trondhjemite and granodiorite (TTG). Along its northern boundary, at a distance of approximately

15 km from Montagne d'Or, the PGB is bounded by sandstones and conglomerates of the NGT. Along the southern margin, the Greenstone Belt is in contact with large intrusive domes of TTG.

The PGB is locally limited to the south and west by regionally extensive post-orogenic granites and to the east by Inferred high-grade metamorphic rocks of migmatitic and granitic gneiss. To the north, a narrow band of Paramaca-Armina Formation is unconformably overlain by the Upper Detrital Series (Ensemble Detrique Superieur EDS), silici-clastic sediments comprised of the Bonidoro, Orapu and Rosebel Formations. The EDS are surrounded by gabbro and granite and are interpreted as having been deposited in pull-apart basins associated with the NGT.

The felsic-mafic metavolcanic rocks of the PGB are overlain by the Armina Formation, a series of alternating sedimentary rocks (sandstones, graywackes and pelites); however, this formation has not been intersected by drilling in the Project area. The BRGM obtained a radiometric age in the Project area of 2,152 ± 8 Ma from a rhyolite which provides a possible date for the volcanic series however the age of the mineralization is unknown. Locally, gabbro intrusions occur which have yielded radiometric dates of 2,150 Ma to 2,145 Ma, similar to the TTG.

The PGB and EDS are probable equivalents or correlatives of respectively the Birimian and Tarkwaian sedimentary sequences of the West African Shield and may have been co-extensive prior to the separation of Gondwanaland in the Mesozoic (Figure 7.1.3). The Paul Isnard Project lies within the northern PGB and is comprised of mafic and felsic metavolcanic rocks of the Paramaca Formation.



Source: Columbus, 2015

Figure 7.1.3: Map Showing Correlation of the Guiana Shield with the West African Birimian Shield

7.2 Property Geology

7.2.1 General

Montagne d'Or occurs within a bimodal felsic-mafic series of Proterozoic volcanic rocks. The gold mineralization is hosted within a 400 m thick, tightly to isoclinally folded sequence of predominantly felsic and lesser mafic volcanic rocks. The units strike east-northeast, dip steeply south and are exposed on the northern slopes of Dékou-Dékou Mountain.

The eastern portion contains dominantly mafic volcanics with only minor amounts of felsic volcanics. The mineralized units have been strongly deformed, as evidenced by a penetrative S1 foliation that locally transposes S0 and in places is mylonitic. The orientation of the S1 foliation is constant throughout the section, striking on average 084° with an average 72°S dip. The intensity of deformation varies significantly over the distance of a few meters. The deposit is cross cut by post deformation diabase dikes.

The volcanic complex of Montagne d'Or is bounded in the north by granite and gneiss and is bounded along its southern margin by amphibolites that were thrust over the volcanic rocks. A sliver of detrital metasedimentary rocks is locally wedged beneath the overthrust amphibolites. The metavolcanic rocks have metamorphosed to greenschist grade.

The entire region has undergone Tertiary age lateritic weathering which resulted in a saprolite cover of varying thickness and in which variable lateral movements have taken place.

7.2.2 Lithology

The Montagne d'Or deposit is hosted within a tightly to isoclinally folded, steeply south dipping lithological package consisting of felsic and mafic metavolcanic rocks that are assigned to the PGB. The mafic metavolcanic rocks were previously divided into two units, a Lower Mafic Unit that lay to the north of the deposit and an Upper Mafic Unit that comprised the eastern part of the deposit (Coffey, 2014). Here, a single mafic metavolcanic unit is interpreted (Figure 7.2.2.1). The grouping of both of the previously defined mafic units into a single unit is justified by the paucity of data that are available for the region to the north of the deposit. The metavolcanic package is intruded by three distinct felsic to intermediate plutonic units that host only minor amounts of gold; from oldest to youngest these are granodiorite, quartz-feldspar porphyry and feldspar porphyry. Quartz-carbonate veins occur throughout the deposit but do not contain significant mineralization.

To the north of the deposit, the metavolcanic rocks are bounded by granite. On the southern side of the Montagne d'Or deposit, the metavolcanic host rocks are structurally overlain by a metasedimentary package consisting of quartzites, black shales and pelitic and graphitic schists. That metasedimentary package is in turn structurally overlain on its southern side by an amphibolite unit.

The metavolcanic and metasedimentary units underwent greenschist grade peak metamorphic conditions. Whole-rock geochemistry data show that the felsic lithologies have a calc-alkaline chemistry and were likely deposited in an arc or back-arc basin environment. Whole rock compositions range between granite and granodiorite (Suter, 1999; GoldFields, 2001).

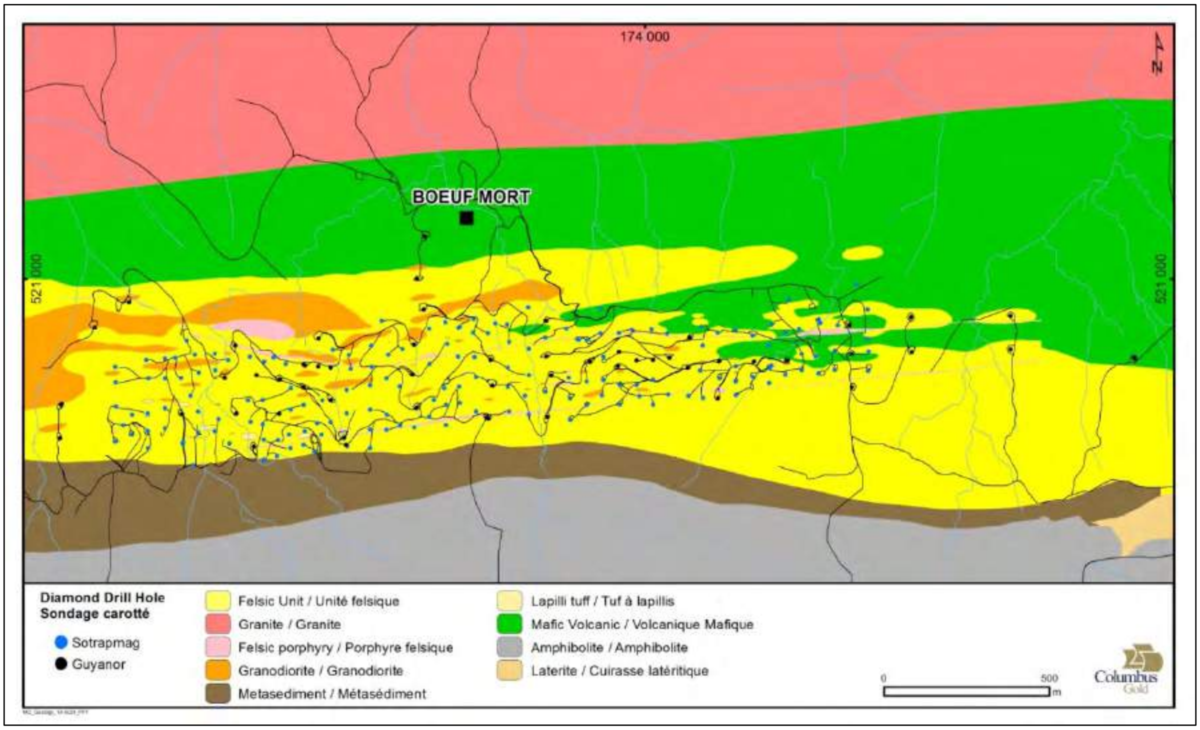
All units described above are cross-cut by a series of northeast striking diabase dikes.

Over 80% of the mineralization at the Montagne d'Or deposit is hosted by felsic metavolcanic units, mainly the Felsic tuff unit as defined here.

The tightly folded metavolcanic and plutonic rocks that represent the geology of the deposit can be assigned to the following principal units, listed from oldest to youngest, and that are described in the following paragraphs:

- Mafic metavolcanics;
- Felsic tuff;
- Lapilli tuff;
- Granodiorite;
- Quartz-feldspar porphyry; and
- Feldspar porphyry.

A schematic of the local geology of the Montagne d'Or prospect is shown in Figure 7.2.2.1.



Source: Columbus, 2015

Figure 7.2.2.1: Schematic Overview of Main Local Geological Units

Mafic Metavolcanics

This unit occurs predominantly in the eastern portion of the deposit where it is tightly infolded with the felsic tuff unit. The mafic metavolcanics may locally be stratigraphically intercalated with the felsic tuffs. The rocks consist of alternating sequences of mafic flows, intermediate to mafic tuffs and mafic dikes. The flows are generally non-schistose, fine grained, massive, locally feldspar phyric, weakly to moderately magnetic, and dark-grey to black in color. Locally observed vesicular and hyalopilitic zones have been interpreted as evidence for a flow origin for the bulk of the unit.

The mafic dikes that are included in this unit are very fine grained and slightly chloritized along their margins. The dike contacts are slightly oblique to schistosity. They are deformed, indicating emplacement early in the geological evolution of the deposit and they are thought to represent synvolcanic dikes and sills petrogenetically related to mafic flows. The dikes have very poor lateral continuity.

The mafic metavolcanic unit may represent part of a bimodal volcanic complex that could include the felsic extrusive units or they may be part of an older crustal section upon which the felsic tuff and the Lapilli tuff would have been deposited. Ongoing geochemical studies should provide more information on the petrogenetic origins of the different metavolcanic units.

Felsic Tuff

The felsic tuff unit consists predominantly of rhyolitic to dacitic rocks many of which preserve a fine lamination that suggests an origin as pyroclastic deposits. It is likely that rhyolitic and dacitic flows also make up a significant proportion of the unit. The groundmass is essentially quartz, feldspar and sericite. The rock is light grey in color and it is generally strongly foliated. Quartz phenocrysts represent up to 10% and they often preserve euhedral bipyramidal shapes. The phenocrysts are embedded in a holocrystalline matrix of fine-grained quartz-feldspar-biotite-(sericite-chlorite). Primary magnetite is often lacking. Locally the quartz phenocrysts are flattened and stretched, with a distinctive blue tint. Pressure shadows at the tips of the deformed phenocrysts may be filled with fibrous quartz and / or sulfide minerals, principally pyrite.

Over 80% of the mineralization at the Montagne d'Or prospect is hosted in the felsic tuff unit.

Lapilli tuff

The Lapilli tuff unit consists of rocks of similar composition to the felsic tuff unit but with quartzo-feldspathic masses (lapilli) hosted within the rhyolitic to dacitic rock matrix. The bulk of the Lapilli tuff unit occurs in the southern part of the Montagne d'Or deposit, close to the sheared contact with the metasedimentary unit. Franklin (1999) suggested that a "felsic lapilli tuff" unit would represent a coarse basal sequence of an ash flow tuff sequence.

Granodiorite

The Granodiorite unit is composed of variably deformed, medium to coarse grained rock the main constituents of which are quartz-feldspar-biotite. Much of the unit is more or less equigranular although sub-rounded quartz and euhedral feldspar phenocrysts are common and are sometimes enclosed within a finer grained groundmass giving a porphyritic texture. The rock is light gray but locally is has a gray to cream color due to sericitization and possibly some albitic and silicic alteration as well. Where the rock is strongly altered the primary texture is largely obliterated.

Quartz-feldspar Porphyry

This unit has a mineralogy that is similar to the Granodiorite unit from which it differs in color and texture. The Quartz-feldspar porphyry is light gray to white and contains a large proportion of euhedral to subhedral phenocrysts of both quartz and feldspar. This unit might be a porphyritic facies of the Granodiorite unit; however, it tends to form homogeneous intervals of several meters in drill core and it is here assigned to its own unit.

Feldspar Porphyry

The Feldspar porphyry unit forms two dikes that are documented to cross-cut the Mafic volcanic, Felsic tuff and Granodiorite units. The rock is of intermediate to felsic composition with a dark grey color and abundant, euhedral to subhedral feldspar phenocrysts. The rock can also contain a small proportion of blue quartz phenocrysts locally. The texture is invariably porphyritic and it can be strongly sheared, suggesting the dikes may have been emplaced within active shear zones.

Quartz-carbonate Veins

Quartz-carbonate veins vary in thickness from the millimeter to meter scales. They are observed to cross cut the principal tectonic S1 foliation and are also deformed and folded, hence they are interpreted to have formed syn-orogenically. The veins are not generally associated with mineralization. Within mafic flows and intrusions, they occur as white, meter scale veins that cross-cut lithologic layering. The quartz veins within the felsic units are thin and are white or blue-grey in color.

Alteration

Gold mineralization at the Montagne d'Or deposit is accompanied by pervasive alteration which includes sericite, secondary biotite (generally retrograded to chlorite) and secondary K-feldspar with locally associated quartz. Alteration products are the result of partial replacement of all lithologies due to reactions with the Fe and sulfide rich mineralizing fluids. The predominant additions to the rock geochemistry were sulfur and iron, as well as potassium, gold, and base metals, with a concomitant removal of sodium and calcium. The precipitation mechanism for gold was likely direct interaction of hydrothermal fluids with the country rocks.

Sericite is the dominant alteration phase in the shallower part of the drillholes, from approximately 40 to 120 m down-hole depth. It transitions into secondary biotite below 150 m. The most pervasive alteration is dominantly a phyllic assemblage. This includes quartz-sericite-pyrite and veinlet-controlled potassic assemblages of secondary biotite, and associated pervasive secondary K-feldspar. A less common, propylitic assemblage consists of chlorite-epidote-calcite. Veinlet assemblages include; quartz-pyrite-pyrrhotite-chalcopyrite, secondary biotite-pyrite-pyrrhotite, and magnetite-pyrrhotite-chalcopyrite-quartz-chlorite with minor amounts of red garnet. Chloritization occurs as a pervasive alteration of mafic units, and as millimeter-scale veinlets within felsic lithologies. The chlorite is Fe-rich, in contrast to Mg-rich chlorite typically associated with VMS type alteration. There is no documented correlation between chloritization and gold content. There is, however, a weak correlation between "hyperchlorite" zones and gold mineralization. The hyperchlorite zones are typically deficient in gold but commonly located adjacent to strongly auriferous zones. The prominent addition to the mafic rocks is Fe^{3+}, as well as gold. This is in part due to addition of sulfide, and perhaps to formation of Fe-rich chlorite. The addition of K_2O, as either sericite, secondary biotite, or secondary K-feldspar is also present. Alteration is typically strongest at the margins of the mineralized zones.

Chlorite alteration within mafic and intermediate units may include some secondary biotite. Zonation of peripheral Pb-Zn disposed about an Au-Cu center is also suggestive of a porphyry-type system. Late stage, narrow quartz veins are planar and cross cut the foliation and mineralized veinlets. They typically have a broad selvage of carbonate-chlorite alteration.

Hyperchlorite alteration zones at Montagne d'Or are composed of variably chloritized portions of nearly all lithologies. They occur predominantly in the mafic volcanic units, intermittently in the felsic units and rarely in mafic intrusive units. The mineralogical and textural characteristics of the zones are quite similar in both mafic and felsic units. The hyperchlorite alteration zones are composed of well foliated biotite (with incipient chlorite replacement), and locally contain a calc-silicate-rich assembly of actinolite, garnet, quartz, calcite-dolomite and magnetite + pyrite, chalcopyrite and pyrrhotite. The magnetite within this assemblage appears to be hydrothermal, and some magnetite rich intervals with sulfides can be highly auriferous. These zones are interpreted as reflecting primary mineralization as opposed to post-mineralization processes.

The edges of the felsic tuff unit are characterized by chlorite veining. Quartz phenocrysts are preserved while most of the primary textures are destroyed, particularly within central parts. Sulfide rich zones up to 50% can be associated with the chloritic alteration. Some rocks logged as mafic tuff may actually represent highly chloritized felsic lithologies. Visual discrimination of hydrothermal and metamorphic chlorite is very difficult.

Silicification is fairly pervasive in all volcanic units. Within the center of the Montagne d'Or prospect, less silicified units tend to have a higher sulfide content.

Sericitization is a major and widespread alteration feature within the felsic units. It has been interpreted as a later overprinting alteration stage on an earlier secondary K-feldspar. There is no documented association between sericitic alteration and gold content. However, the early BRGM regional geochemistry showed that K and Ba are elevated proximal to faults and shear systems. This feature in time provided the pathfinder to the Montagne d'Or prospect gold mineralization.

Carbonate alteration occurs within felsic rocks as fine stringers and replacements. Within mafic units, calcite development is more pervasive, occurring as massive replacement within rhythmically banded tuffs, and as carbonate-chlorite or quartz-carbonate veinlets. It is difficult to separate the hydrothermal alteration carbonates from that derived by regional metamorphic processes. No correlation has been noted between carbonate alteration and gold content.

7.2.3 Structure

The Paul Isnard Project area has experienced two distinct deformational events. The first involved ductile deformation during the Lower Proterozoic accretionary arc tectonism that formed the Guiana Shield. The second is a more brittle deformation event associated with the faulting within the NGT.

The first phase of regional deformation was associated with a regional northeast-southwest compression that led to the development of the pervasive S1 schistosity that strikes 080° to 100° and that dips steeply south. At the Montagne d'Or deposit, the average strike of S1 is 084° and the average dip is 72°S. This principal deformation event postdates mineralization as evidenced by the highly deformed sulfide fabric. However, at the Montagne d'Or, the crystallization of sulfides with pressure shadows at the tips of deformed phenocrysts indicates that some sulfide may have been remobilized during the tectonic event or that a second sulfide deposition event may have been syn-deformational.

Regionally, the development of the S1 schistosity was accompanied by Upper Greenschist Facies and Lower Amphibolite Facies metamorphism, locally associated with the emplacement of granitic plutons and migmatization. At the Paul Isnard project, S1 is associated with the deformation event that resulted in the very tight to isoclinal folding of the Montagne d'Or deposit and also in the thrusting of the amphibolite unit over the deposit.

The second phase of regional deformation postdates the EDS sediments and is related to sinistral transcurrent tectonism, marking the contact between the NGT and PGB. As a result of the second deformation, the earlier S1 schistosity is locally crenulated. A weak S2 fabric is characterized by a spaced cleavage, which strikes 060°. At the Montagne d'Or deposit, late diabase dikes have a preferred strike orientation between 060° and 065°, sub-parallel to S2, suggesting they were emplaced with shears, faults or fractures that had formed during the transcurrent tectonic event.

Regionally, a well-developed set of faults and fractures with four principal orientations were also developed and these may also be expressed at the scale of the Paul Isnard project. The relative intensity of these brittle structures listed from strongest to weakest are:

- North-south (48%);
- Northeast-southwest (28%);
- Northwest-southeast (16%); and

- East-west (7%).

7.3 Mineralization

The Montagne d'Or prospect consists of a family of tabular mineralized bodies that form closely-spaced sub-parallel east-northeast (084°) striking and steeply (72°) south-dipping mineralized zones. Mineralization has been encountered over a strike length of more than 2,500 m and to a vertical depth of at least 200 m. Only a small portion of the gold mineralization has been subjected to upper level oxidation. The significant fine-grained gold mineralization is principally affiliated with sulfide veins and masses within fresh country rock that begins at shallow depths.

Historically, on a macroscopic scale, two significant styles of gold mineralization have been recognized although they show a gradational relationship between each other:

- Semi-massive sulfide (SMS) with gold mineralization, and
- Sulfides in disseminated stringers with gold mineralization.

SMS was a term coined by previous operators and was used to support a "VMS" type model for the mineralization. It is characterized by a high sulfide content (>20%) and occurs over intervals ranging from tens of centimeters to up to 4 m. This mineralization was later interpreted to represent zones of thicker, deformed and transposed sulfide ± quartz-rich veins and a denser distribution of disseminated sulfide as compared to that of the disseminated type.

The SMS also includes sulfide-rich breccia dykes, which host rolled and milled clasts of host rock within a ductily deformed pyrite-chalcopyrite-pyrrhotite matrix. In addition, bornite is present, and minor amounts of arsenopyrite have been identified petrographically. There is a clear correlation between sulfide veinlets and sulfide-rich breccia zones and high gold grades. Relatively minor amounts of total sulfide (i.e., disseminated + vein and veinlet + breccia – hosted sulfide representing 2% to 5% total rock volume), locally resulting in significant although erratic, high gold concentrations, commonly attain values of tens of grams per tonne gold over standard 1 m sample intervals.

Disseminated mineralization is characterized by the presence of finely disseminated to finely fracture controlled sulfides, chiefly pyrite but with lessor and locally important chalcopyrite and pyrrhotite.

Close inspection of core and outcrop indicate that gold associated with this style of mineralization is in part controlled by the abundance of fine sulfide-quartz veinlets and fracture fillings which have been strongly (isoclinally) folded, sheared and transposed parallel to the S1 fabric. Grades for this mineralization type are dependent upon disseminated sulfide and sulfide-quartz veinlet density, but are generally low, in the 0.5 g/t Au to 3 g/t Au range over sample intervals which average approximately 1 m in length.

Mineralization is hosted by felsic, mafic and intercalated mafic/felsic rocks to varying degrees. However, approximately 80% of the gold mineralization in the deposit occurs within the more felsic units, mainly the Felsic tuff unit.

The mineralization appears as elongated lenses of higher grade material within broader zones of low grade but anomalous mineralization (0.25 g/t Au to 0.4 g/t Au). Several distinct anomalous mineralized domains are recognized, separated by barren intercalated mafic and felsic rocks.

Disseminated sulfide mineralization is hosted mainly within the Felsic tuff unit and is predominantly or entirely pre-orogenic. Disseminated pyrite crystals are coarse and also locally stretched. Some mafic units carry similar mineralization but with a notably lower sulfide vein density.

The Montagne d'Or deposit is now thought to be part of a stratiform/stratabound deposit type. Mineralization consists of pyrite, pyrrhotite and chalcopyrite with minor sphalerite, magnetite and arsenopyrite. Arsenopyrite, although observed, does not appear to have an obvious relationship with either gold or copper mineralization. Distinct phases are reported as stratiform disseminated sulfides, stockwork sulfide veinlets and layers of semi-massive sulfides that are tectonically transposed. The latter facies is considered as syn-volcanic in origin and as the most favorable occurrence for gold mineralization.

The disseminated sulfide veins could be related to feeder zones and/or remobilized on fold hinges and shear zones. In addition, evidence is found for tectonic remobilization with sulfides concentrated within fold hinges and pressure shadows, and cross-cutting sulfide-bearing veins.

Visible gold occurs in chlorite-rich zones or is spatially related to sulfide mineralization (after Giraud, Tremblay, Jébrak and Lefrançois, 2014). Figure 7.3.1.1 shows a photograph of native gold hosted by mafic volcanic rocks in drillhole MO1266 at a depth of 245 m. This particular one meter interval ran 80.75 g/t Au. There is generally an increase in gold grades as sulfide (excluding pyrrhotite) content increases. Microscopic studies indicate that gold occurs as very fine grains in the host rock groundmass and at the junctions of quartz crystals. Gold has only very rarely been seen as inclusions within sulfide minerals.

Source: Columbus, 2013

Figure 7.3.1: Example of Visible Gold Occurring within Mafic Volcanics (MO1266)

8 Deposit Type

The current interpretation is that Montagne d'Or is a deformed volcanogenic massive sulfide deposit (Ross 2014). Ross based this interpretation largely on the following details of the deposit.

- The presence of pillow basalts in the Upper Mafic Unit, making at least this part of the volcanic succession submarine, and formed on the ocean floor;
- The Felsic Unit is cut by tholeiitic mafic dikes related to the Upper Mafic Unit, whereas the Upper Mafic Unit is cut by calc-alkaline QFP dikes related to the Felsic Unit;
- This means that the Felsic Unit and the Upper Mafic Unit are broadly contemporaneous; by association, the Felsic Unit is therefore also submarine;
- The Felsic Unit is indeed, partly, a layered volcaniclastic pile (Franklin et al., 2001). There are some QFP intrusions in this pile (as noted by Shaw, 2001), but at least some of the felsic rocks were deposited on the sea floor (Franklin, 1999); volcaniclastic rocks are ideal for sub-seafloor replacement;
- Alteration mineralogy is dominated by sericite and chlorite, which are typical VMS minerals, or their metamorphosed equivalents (e.g., garnet, biotite); and
- The sulfides were emplaced before tectonic deformation.

A submarine volcanic arc is presently thought to be the likeliest setting for the formation of the Montagne d'Or deposit; the Izu-Bonin arc south of Japan may be a plausible analogue (there are seafloor massive sulfides deposits currently forming in this arc; e.g., Glasby et al., 2000). A back-arc with a strong subduction signature is also possible, as back-arc basins can have voluminous felsic magmatism too, for example the Manus Basin offshore Papua New Guinea, where there are also seafloor massive sulfides actively accumulating (e.g., Binns and Scott, 1993; Paulick et al., 2004; Ross, 2014).

9 Exploration

Since completing the previous technical report effective to the end of June 2014, Columbus has only conducted exploration drilling. The latest drilling program was completed in November 2014.

10 Drilling

Sections 10.1 and 10.2 have been excerpted from the Coffey 2014 Technical Report. Section 10.3 is updated current to this report. Standardizations have been made to suite the format of this report.

Since the inception of exploration by Columbus, a total of 171 drillholes (MO1361 to MO14231) have been completed testing the Montagne d'Or deposit.

Earlier drilling completed by Guyanor consists of a total of 56 drillholes (MO9601 to MO9856) totaling 10,916 m on from 1996 to 1998. Assays from these drillholes are of lower quality (a characteristic that has been taken into account during resource classification) but are considered as relevant and fit-for- purpose for the resource estimate. (note: all holes drilled by Columbus are within the deposit; however, there are three Guyanor holes, hole numbers MO57, MO58, and MO59, which were drilled in 2001 on the Apollon target located to the southeast of the deposit, and drillhole MO60, the only hole drilled in 2007, which is not included in the database as it is a twin of a previous hole).

10.1 Guyanor Drilling Program: 1996 to 1998

From 1996 to 1998, Guyanor completed a total of 56 drillholes (MO9601 to MO9856) totaling 10,916 m on the Montagne d'Or prospect. Drilling was done under contract by Major Drilling Company of Canada. Drill pads and access were prepared using bulldozers and/or excavators; every attempt was made to limit deforestation and for this reason, use of an excavator was preferred for construction of drill pads.

Drilling procedures were to collar each hole with HQ bits (core diameter 6.35 cm) and reduce to NQ (core diameter 4.76 cm) when hard and not oxidized rock was intersected. Core recovery was routinely measured and recorded for each core run. Core recoveries overall were generally excellent. Major Drilling used Longyear 38 wireline diamond drilling rigs. Drillhole spacing is variable, from 50 to 250 m. Drill fences are spaced 100 to 200 m apart. The presence of clearly visible, regionally consistent, and well-defined S_1 fabric allowed the core to be manually oriented in the core boxes, although local variations have, on occasion, caused incorrect orientation. Saprolite was not oriented due to the absence of a clearly defined fabric. Core was placed in plastic core boxes at the drill site, with core markers placed at the start and end of each core run, and boxes securely covered. Core boxes were transported back to camp for detailed logging and core splitting. Core photography was carried out infrequently. All drillhole collars were a surveyed for X, Y, Z coordinates tied to the mine grid shortly after completion so as to provide an accurate location for resource estimation. The mine grid was tied to the X, Y UTM grid and the Z coordinates were shifted 1,000 m above mean sea level so that no negative elevations were present within the drillholes. Drillhole location surveys were performed by Guyanor survey crews and external surveyors from SATTAS using TDS equipment.

The first 47 drillholes were surveyed downhole for deviation and deflection by Major Drilling, mainly using acid bottle etch or Pajari /Tropari mechanical instruments. Downhole survey intervals were at 50 m. The final eight drillholes were surveyed in with Sperry Sun equipment. The downhole surveys using acid bottle etch and Tropari equipment were criticized within internal Guyanor documents as poorly suited to the task as only dip and no azimuth is recorded. The inaccuracy of the early downhole surveys is considered in Mineral Resource classification although it should be noted that due to the relatively short length, significant drillhole deviation and deflection at Montagne d'Or are minimal, with deflection of 5° to 10° over 200 m typical. Four drillholes were not collar surveyed; however, the planned hole coordinates have been used. Details for the drilling completed by Guyanor from 1996 to 1998 (56 holes in total) are listed in Table 10.1.1.

Table 10.1.1: Drillholes (56 in Total) Drilled by Guyanor from 1996 to 1998

Drillhole	Easting	Northing	Elevation	Azimuth	Dip	Depth (m)	Operator	Year
MO9601	173091.8	520520.8	260.89	0	-60	199.8	Guyanor	1996
MO9602	173096.5	520499.6	268.60	0	-60	52.5	Guyanor	1996
MO9603	173051.7	520634.9	220.88	0	-57	271.6	Guyanor	1996
MO9604	173311.3	520611.1	269.68	0	-61	208.6	Guyanor	1996
MO9605	173298.7	520711.7	229.45	0	-61	201.3	Guyanor	1996
MO9606	173706.1	520583.6	273.74	0	-60	199.6	Guyanor	1996
MO9607	173717.2	520708.8	258.57	0	-60	202.6	Guyanor	1996
MO9608	173703.7	520765.8	227.79	0	-60	199.6	Guyanor	1996
MO9609	173703.5	520873.9	180.19	0	-60	199.6	Guyanor	1996
MO9610	173331.9	520908.4	173.50	0	-60	199.6	Guyanor	1996
MO9611	173302.2	520802.4	191.95	0	-63	201.6	Guyanor	1996
MO9612	173014.3	520820.4	163.98	0	-61	201.55	Guyanor	1996
MO9613	172973.3	520738.8	182.53	0	-60	59.7	Guyanor	1996
MO9614	172969.8	520742.4	182.30	358	-61	205.6	Guyanor	1996
MO9615	172763.0	520800.2	186.52	0	-59	193.6	Guyanor	1996
MO9616	172730.8	520700.8	189.25	0	-60	199.6	Guyanor	1996
MO9617	173335.5	521128.7	120.17	0	-60	151.6	Guyanor	1996
MO9618	173312.4	521000.7	151.73	0	-60	156.6	Guyanor	1996
MO9719	174129.7	520732.4	296.82	0	-60	199.5	Guyanor	1997
MO9720	174136.4	520822.6	247.89	0	-60	200	Guyanor	1997
MO9721	173540.1	520678.7	273.57	0	-60	200	Guyanor	1997
MO9722	173534.4	520755.3	237.60	0	-60	199.5	Guyanor	1997
MO9723	172233.0	520519.2	233.78	0	-60	199.5	Guyanor	1997
MO9724	172236.9	520619.4	219.33	0	-60	198.5	Guyanor	1997
MO9725	172766.0	520594.6	228.71	0	-60	199.5	Guyanor	1997
MO9726	174626.3	520774.4	204.39	0	-60	199.5	Guyanor	1997
MO9727	174619.1	520860.7	184.97	0	-60	199.5	Guyanor	1997
MO9728	174225.2	520750.7	300.96	0	-60	199.5	Guyanor	1997
MO9729	172337.5	520852.9	172.76	0	-60	202.6	Guyanor	1997
MO9730	172441.8	520929.5	141.84	0	-60	199.6	Guyanor	1997
MO9731	172897.2	520696.3	208.28	0	-60	199.6	Guyanor	1997
MO9732	172819.0	520493.9	251.18	0	-60	277.6	Guyanor	1997
MO9733	172601.1	520591.9	231.09	0	-60	199.6	Guyanor	1997
MO9734	173522.6	520581.7	321.30	0	-60	22.7	Guyanor	1997
MO9735	173528.4	520578.9	321.39	1	-61	295.6	Guyanor	1997
MO9736	173919.9	520736.5	285.07	0	-60	199.6	Guyanor	1997
MO9737	174222.9	520641.5	298.81	0	-60	271.6	Guyanor	1997
MO9738	174430.2	520753.1	262.39	0	-60	263.9	Guyanor	1997
MO9739	174627.0	520672.8	218.79	0	-60	249.6	Guyanor	1997
MO9740	172969.6	520672.7	227.21	0	-59.5	229.6	Guyanor	1997
MO9741	173051.5	520732.8	177.60	0	-60	196.6	Guyanor	1997
MO9742	173013.2	520736	179.61	358	-60	190.6	Guyanor	1997
MO9743	174806.0	520885	203.38	0	-60	187.6	Guyanor	1997
MO9744	174808.3	520780.7	209.90	0	-60	199.6	Guyanor	1997
MO9745	175107.2	520887.9	193.47	0	-60	193.6	Guyanor	1997
MO9746	175107.0	520788.2	203.67	0	-60	238.4	Guyanor	1997
MO9747	175479.7	520760.9	184.40	90	-60	120.06	Guyanor	1997
MO9748	175479.7	520760.9	184.40	0	-60	193.6	Guyanor	1997
MO9849	172826.3	520709.3	205.80	0	-60	178.6	Guyanor	1998
MO9850	174331.3	520751.2	296.62	0	-60	150.9	Guyanor	1998
MO9851	174025.5	520755.9	277.46	0	-60	199.6	Guyanor	1998
MO9852	173923.2	520780.7	266.02	0	-60	151.6	Guyanor	1998
MO9853	173834.6	520751.4	257.74	0	-60	190.6	Guyanor	1998
MO9854	172895.1	520592.8	260.41	0	-60	199.6	Guyanor	1998
MO9855	173975.4	520754.1	277.44	0	-60	202.6	Guyanor	1998
MO9856	174075.4	520762.1	270.31	0	-60	211.6	Guyanor	1998

Source: Coffey, 2014
Coordinate System: CSG 167 datum UTM Zone 22

10.2 Columbus Drilling Program: 2011 to 2012

From the end of 2011 until August 2012, Columbus drilled 45 drillholes (MO11061 to MO12105) totaling 15721.45 m, named as Phase I of Columbus drilling. Drilling was done under contract by Performax Drilling of Val d'Or, Quebec, Canada.

Drilling procedures were very similar to those in the previous dill programs. All drillholes were collared using HQ equipment, downsizing to NQ after intersecting solid generally un-oxidized rock. Core recovery at the drill site averages 87.5% in HQ core (saprolite zone) increasing to 99.6% in NQ core (fresh material). Performax used a containerized Longyear 38 drill.

The drill program was designed to provide infill drillholes in known mineralized areas and to continue exploring strike extensions of the mineralization. Drillhole spacing in the central part of the mineralized zone varies between about 35 and 75 m and 100 to 200 m on the extremities.

The drillholes are, in general, inclined moderately to the north whereas the mineralization dips at 68° to 72° to the south. Therefore, the drillholes intercepts do not represent true thickness but true thickness averages approximately 75% of the intercept distance. Down-hole surveying of the drillholes was performed by the drill crew using a Reflex instrument. In some cases the Reflex instrument did not function correctly. For these holes an average was taken of measurements from 10 holes and these values were used where data could not be measured. Given that the deviation in all of the drillholes is very consistent this method is considered acceptable with minimal risk to the resource estimate.

A private contractor was hired to undertake surveying of all collars for holes MO1161 to MO11105 using CGS1967 datum. All drillhole collars were surveyed using GPS Total Station equipment. All previous drillhole coordinates were converted to CGS 1967 format, the 1,000 m elevation addition removed that was present in the earlier data and four older drill collars checked by re-surveying.

Details for the drilling completed by Columbus from 2011 to 2012 (45 in total) are provided in Table 10.2.1.

Table 10.2.1: Drillholes (45 in total) completed by Columbus (Phase 1) in 2011 and 2012

Drillhole	Easting	Northing	Elevation	Azimuth	Dip	Depth (m)	Operator	Year
MO11061	173870.5	520648.2	299.21	0	-70	350	Columbus	2011
MO11062	173984.3	520647.0	318.16	0	-70	399.3	Columbus	2011
MO11063	174072.9	520652.0	323.75	2	-60	378.5	Columbus	2011
MO11064	172972.3	520539.8	287.46	2	-60	419	Columbus	2011
MO11065	172891.8	520514.1	272.39	0	-60	356	Columbus	2011
MO12066	173770.8	520701.9	260.19	0	-60	329	Columbus	2012
MO12067	173637.9	520647.7	267.48	0	-60	361	Columbus	2012
MO12068	173441.4	520625.4	302.83	0	-60	380	Columbus	2012
MO12069	172745.1	520503.5	256.82	0	-60	257	Columbus	2012
MO12070	173025.8	520633.6	231.30	0	-60	302	Columbus	2012
MO12071	173206.0	520874.6	177.80	180	-50	308	Columbus	2012
MO12072	173057.3	520786.8	182.46	180	-50	350	Columbus	2012
MO12073	172615.6	520814.2	197.01	180	-50	440	Columbus	2012
MO12074	174676.1	520781.4	214.08	0	-60	275	Columbus	2012
MO12075	174516.8	520766.1	220.82	0	-60	251	Columbus	2012
MO12076	174435.1	520938.4	197.49	180	-50	322	Columbus	2012
MO12077	174641.4	520982.6	175.94	180	-50	429	Columbus	2012
MO12078	173868.8	520909.9	204.46	180	-50	411	Columbus	2012
MO12079	173647.8	520914.1	180.94	180	-50	375	Columbus	2012
MO12080	173438.0	520852.2	203.50	180	-50	387	Columbus	2012
MO12081	174275.9	520736.9	306.87	0	-60	345	Columbus	2012
MO12082	174168.4	520723.3	307.91	0	-60	351	Columbus	2012
MO12083	174377.1	520732.0	282.60	0	-60	317	Columbus	2012
MO12084	174383.6	520739.2	282.61	180	-50	152	Columbus	2012
MO12085	174131.7	520647.2	332.27	0	-60	425	Columbus	2012
MO12086	174177.0	520640.8	324.31	0	-60	425	Columbus	2012
MO12087	173436.6	520764.9	239.73	0	-60	302	Columbus	2012
MO12088	173485.4	520764.4	247.61	0	-60	299	Columbus	2012
MO12089	173586.3	520732.8	244.44	0	-60	299	Columbus	2012
MO12090	173303.8	520552.2	287.75	0	-60	409	Columbus	2012
MO12091	173220.9	520589.5	273.50	0	-60	400	Columbus	2012
MO12092	173022.7	520529.7	286.08	0	-60	374	Columbus	2012
MO12093	172924.8	520529.8	281.68	0	-60	448	Columbus	2012
MO12094	173101.5	520495.5	269.14	0	-60	464	Columbus	2012
MO12095	172845.4	520562.1	264.26	0	-60	365	Columbus	2012
MO12096	172604.5	520508.4	237.74	180	-60	119	Columbus	2012
MO12097	172603.7	520503.0	238.06	0	-60	422	Columbus	2012
MO12098	172636.2	520437.3	239.94	0	-60	389	Columbus	2012
MO12099	172423.6	520558.3	301.23	0	-60	221	Columbus	2012
MO12100	173169.6	520544.9	282.64	0	-60	381	Columbus	2012
MO12101	173261.1	520557.3	283.30	0	-50	350	Columbus	2012
MO12102	173363.8	520634.2	274.71	0	-60	344	Columbus	2012
MO12103	173394.2	520670.0	272.00	0	-60	281	Columbus	2012
MO12104	173490.1	520704.8	273.95	0	-70	346.65	Columbus	2012
MO12105	173587.3	520673.7	273.08	0	-60	413	Columbus	2012

Source: Coffey, 2014
Coordinate System: CSG 167 datum UTM Zone 22

10.3 Columbus Drilling Program: 2013 to 2014

From early 2013 until November 2014, Columbus drilled a total of 126 drillholes (MO13106 to MO14231) (25,073.6 m) and 13 abandoned and re-drilled holes (495.0 m), for a total of 25,568.6 m. This corresponds to the Phase II of Columbus drilling. Drilling was done under contract by Performax Drilling of Val d'Or, Quebec, Canada. Drilling procedures were the same to those in the previous programs. All drillholes were collared using HQ equipment downsizing to NQ after intersecting solid generally un-oxidized rock. Core recovery at the drill site averages 87.5% for HQ drillholes in the saprolite zone and 99.6% in NQ drillholes in fresh material. Details of the most recent drillholes completed by Columbus in 2013 and 2014 are presented in Table 10.3.1.

Table 10.3.1: Drillholes (126 in total) completed by Columbus (Phase 2) in 2013 and 2014

Drillhole #	UTM East	UTM North	Elevation (m)	Azimuth	Dip	Length (m)
MO14113	173220	520780	240	0	-60	118
MO14114	173170	520800	195	0	-60	88.6
MO14115	173170	520745	215	0	-60	167
MO14116	173350	520745	230	0	-60	149
MO14117	173260	520665	250	0	-60	161
MO14118	173100	520770	190	0	-60	74
MO14119	173100	520725	190	0	-60	143
MO14120	172930	520760	175	0	-60	122
MO14121	172890	520770	175	0	-60	110.5
MO14122	172810	520765	190	0	-60	111.5
MO14123	172750	520760	180	0	-60	104
MO14124	172700	520760	180	0	-60	101
MO14125	172650	520750	200	0	-60	124
MO14126	172600	520740	210	0	-60	122
MO14127	172500	520760	210	0	-60	98
MO14128	172650	520630	200	0	-60	123
MO14129	173775	520860	200	0	-60	122
MO14130	173875	520820	235	0	-60	98
MO14131	173825	520815	220	0	-60	98
MO14132	173925	520840	240	0	-60	107
MO14133	173975	520835	250	0	-60	121.5
MO14134	174025	520850	235	0	-60	111
MO14135	174075	520840	235	0	-60	131
MO14136	174175	520865	240	0	-60	101
MO14137	174225	520840	255	0	-60	164
MO14138	173590	520865	200	0	-60	116
MO14139	173540	520870	200	0	-60	95
MO14140	174575	520850	190	0	-60	134
MO14141	174675	520895	200	0	-60	119
MO14142	174675	520840	210	0	-60	179
MO14143	174525	520830	215	0	-60	169
MO14144	174475	520850	215	0	-60	158
MO14145	174375	520865	235	0	-60	131
MO14146	174425	520840	230	0	-60	155
MO14147	174525	520880	195	0	-60	101
MO14148	173010	520465	290	0	-60	150.8
MO14149	172850	520630	245	0	-60	164
MO14150	172810	520620	245	0	-60	161
MO14151	172400	520620	275	0	-60	125
MO14152	172500	520600	275	0	-60	161
MO14153	172707	520584	215	0	-52	159.7
MO14154	172650	520730	200	0	-60	155
MO14155	172600	520700	215	0	-60	173
MO14156	172400	520700	230	0	-60	149
MO14157	172500	520700	230	0	-60	184
MO14158A	172700	520720	185	0	-60	22.5

Drillhole #	UTM East	UTM North	Elevation (m)	Azimuth	Dip	Length (m)
MO14158	172700	520720	185	0	-60	153
MO14159	172850	520750	190	0	-60	130
MO14160	173400	520795	220	0	-60	166
MO14161	173700	520835	195	0	-60	158
MO14162	174326	520829	254	0	-60	198
MO14163	174276	520839	247	0	-60	171
MO14164	172399	520510	304	0	-60	266
MO14165	172499	520540	292	0	-60	221
MO14166	172809	520570	249	0	-60	226.6
MO14167	172969	520620	253	0	-60	191
MO14168	172930	520605	270	0	-60	242
MO14169	172849	520515	263	0	-60	287.6
MO14170	172930	520670	225	0	-60	281
MO14171	173051	520568	256	0	-60	257
MO14172	173099	520595	244	0	-60	200
MO14173	173099	520665	215	0	-60	260
MO14174	174025	520695	303	0	-62	316.9
MO14175	173875	520695	290	0	-62	278
MO14176	174025	520620	330	0	-62	365
MO14177	173925	520680	302	0	-62	317
MO14178	173975	520680	307	0	-62	323
MO14179	173925	520620	318	0	-62	329
MO14180A	172969	520500	292	0	-62	125
MO14180	172969	520500	292	0	-62	344
MO14181	173825	520620	299	0	-62	307
MO14182	173775	520575	308	0	-62	344
MO14183A	173775	520640	283	0	-62	98
MO14183B	173775	520640	283	0	-62	15.5
MO14183	173775	520640	283	0	-62	266
MO14184	173700	520660	259	0	-62	293
MO14185	173825	520690	273	0	-62	349
MO14186	172699	520540	230	2	-62	230
MO14187	172650	520600	205	0	-60	299
MO14188	172550	520760	210	0	-60	104
MO14189	172400	520735	219	0	-60	145
MO14190	173875	520765	260	0	-62	177
MO14191A	172499	520490	286	0	-62	62
MO14191	172499	520490	286	2	-62	301
MO14192	172650	520535	220	2	-62	230
MO14193	172550	520610	250	0	-60	308
MO14194	172550	520550	265	0	-60	239
MO14195	172929	520490	273	0	-64	322.8
MO14196	172889	520465	261	0	-62	108
MO14197	172849	520455	259	0	-62	123
MO14198	173650	520590	276	1	-64	320
MO14199A	173440	520675	289	2	-65	30.5
MO14199B	173440	520675	289	2	-65	36.5
MO14199	173440	520675	289	2	-65	320
MO14200	173169	520680	225	1	-63	239
MO14201	173590	520600	301	2	-65	353
MO14206	172550	520700	225	0	-62	191
MO14207	173169	520490	287	0	-62	188
MO14208	173219	520545	293	0	-64	353
MO14209A	173169	520605	264	0.5	-62	15.5
MO14209	173169	520605	264	0.5	-62	247
MO14210	173220	520700	235	0	-62	197

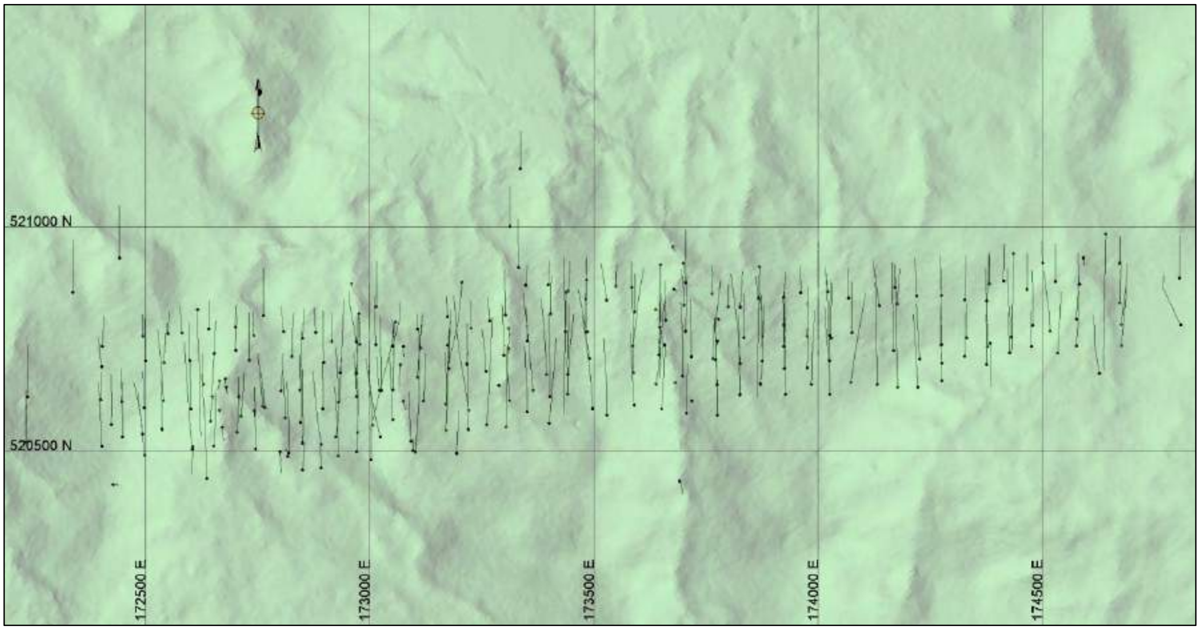
Drillhole #	UTM East	UTM North	Elevation (m)	Azimuth	Dip	Length (m)
MO14211	173650	520710	242	1	-64	293
MO14212	173590	520810	212	1	-63	182
MO14213	173650	520790	199	0	-63	179
MO14214	173825	520815	225	2	-65	194
MO14215A	174225	520700	317	2	-65	18.5
MO14215	174225	520700	317	2	-65	251
MO14216	174576	520784	202	2	-63	210
MO14217	174576	520726	212	1	-63	270
MO14218A	173010	520565	276	2	-63	18.5
MO14218	173010	520565	276	2	-63	269
MO14219	174476	520725	245	1	-63	182
MO14220	174276	520650	286	2	-63	257
MO14221	174275	520690	301	1	-63	227
MO14222	174175	520800	270	2	-64	188
MO14223	174375	520800	265	1	-63	233
MO14224	174476	520774	239	1	-62	239
MO14225A	174326	520705	294	0	-63	6.5
MO14225	174326	520705	294	0	-63	206
MO14226	174526	520716	212	1	-63	280
MO14227A	174376	520685	275	1	-63	15.5
MO14227	174376	520685	275	1	-63	224
MO14228A	174426	520710	258	1	-63	30.5
MO14228	174426	520710	258	1	-63	200
MO14229	174675	520723	216	1	-64	290
MO14230	172450	520590	275	1	-62	143
MO14231	172450	520530	306	1	-63	233
Total Meters						**25,568.6**

Source: Columbus, 2015

10.4 Interpretation of Drillhole Results

The drilling types described above all constitute industry standard methods of exploration for this type of mineralization and material. The sampling procedures all meet industry best practices and an appropriate chain of custody has been utilized during all handling and sampling of the drill core or cuttings. The drillholes are inclined on average at -60° toward the -70° dipping mineralization; therefore, the drillhole intersections do not represent true thickness of the mineralization. The drillholes generally intersect the mineralization at approximately 50°, which SRK considers appropriate to define the geologic model and mineralization.

SRK is of the opinion that best professional judgment, and appropriate exploration and scientific methods were utilized in the collection and interpretation of the drilling data used in this report. The sampling is sufficient and spaced appropriately to support the resource estimation. Figure 10.4.1 presents an overview of the drillhole locations.



Source: SRK, 2015

Figure 10.4.1: Plan View of Drillhole Traces

11 Sample Preparation, Analysis and Security

The information presented in this section concerning pre-2011 sampling and analysis has been largely based on the SRK report, Stryhas, B (2012) with additional information as specified for updated drilling data provided by Columbus.

11.1 Historical Methods

Limited information is available on the historical transport, sampling and analysis of the Guyanor drillholes. The diamond drill core was transported from the drill site to the Boeuf Mort camp where all geologic logging and sampling was conducted. Sample intervals were marked in advance by the Project geologists. The saprolite core was halved with a knife, while fresh rock core was sawn with a powered diamond saw. The original assay lengths range from 0.1 to 4.3 m with an average of 1.0 m. A total of 10,693 samples were taken. The presence of dispersed zones of very narrow sulfide bands, in some cases, forced sample intervals that did not always conform to the actual lithologic breaks. The sawn half-core was bagged, labelled on site, and sent out for assaying.

Sample bags were routinely placed in plastic rice bags and sealed to prevent tampering between the campsite and the laboratory. The remaining half core was returned to the core box and stored for future reference.

Rock quality description (RQD) measurements were completed on selected intervals in seven drillholes during the 1998 campaign. Magnetic susceptibility measurements were completed for 18 drillholes during this campaign (MO9601 to MO9618).

Bulk density measurements on drill core were not performed on a regular basis. The densities used for previous resource estimations utilized bulk densities taken from equivalent or nominal rock types (not described).

The diamond core and channel samples collected in the Montagne d'Or prospect area during the 1996/1998 drilling campaign were dispatched to six separate laboratories for sample size reduction, homogenization, and assay determination. Analytical methodologies utilized were typically fire assayed with an atomic absorption finish. A few samples were assayed by fire assay (FA) with a gravimetric finish. These are appropriate and standard methodologies for gold analysis. There is no documentation in the Project files related to the certification of any of the laboratories used to analyze the Montagne d'Or prospect samples. It was not industry standard of the time to undergo certification procedures.

The QA/QC procedures for the Montagne d'Or Prospect analytical work prior to 1998 utilized check assays performed on quarter core, the remaining half of re-sawn split half core. Most quarter-core samples were collected from barren core (<0.05 g/t Au) and used for blank material. Since the samples were not extracted from the same pulp, the samples are more correctly termed field duplicates. No data are available for assay standards included with any of the drill or channel sample analyses. Internal check assay information is provided for five of the six laboratories that were used for gold assaying.

RSG (2004) provided a review of the QA/QC results obtained during the history of the drilling and they concluded the following: the results of the RSG Global statistical assessment of the quality control data suggest that the SGS Cayenne and CanTech laboratories were producing assay results of an acceptable precision and unknown accuracy, but that the SGS France and Cone Colorado laboratories were not producing assays of an acceptable precision. The various coarse reject check assaying programs indicate that there are serious problem at all or some of the laboratories and that precision levels from all the check assay programs are unacceptable. Correlation between assay pairs is very poor with significant bias shown in some instances. The accuracy of the data produced by each laboratory cannot be assessed without standard reference assay data, and this is a material flaw in the check assay programs completed to that date.

In 2007, Golden Star conducted a modern QA/QC analysis during a re-assay program of the historical drill core at the Paul Isnard deposit. This consisted of re-sampling of the core from a wide distribution of drillholes, insertion of blanks and standards, and submitting all these to an accredited laboratory.

The laboratory employed industry standard sample preparation and the techniques of analyses were appropriate for the level of gold mineralization. The results of the QA/QC verified the credibility of the 2007 re-assay results. This is discussed further in Section 11.3.

11.2 Columbus Drill Program

The following description of sample preparation and core handling protocols applies to all drilling carried out by Columbus to date on the Montagne d'Or prospect. The next sections describe the 2011 and 2012 logging and sampling procedures, which were upgraded for the 2013 and 2014 program (geotechnical logging, core photography, air transport to Cayenne, use of Geotic software, assays on 50 g split by FA AA, assays above 5 g/t Au re-assayed by gravimetrics, refer also Section 12). Program details on the current logging, sampling and QA/QC protocols were discussed in detail with Columbus staff during the site visits by SRK and their systematic application with respect to the Project was confirmed.

11.2.1 Core Logging and Sampling

Drill core is placed in plastic trays at the drill site by the drill crew. Drillers either transport the core to the end of the road for pickup by Columbus personnel or directly to the core shack in the Citron Camp.

Once in the camp the core boxes are opened and placed in order on logging racks within the core logging facility (Figure 11.2.1.1). If space is not available then the core is stored in core racks adjacent to the logging facility.

Source: Columbus, 2015

Figure 11.2.1.1: Core Logging Facility at the Citron Exploration Camp

The drill core is washed to remove any dirt or grease and reconstituted. The core is measured to ensure that there are markers every meter. Basic geotechnical logging is initially undertaken, measuring recovery and RQD.

The core is descriptively logged and marked for sampling by Columbus geologists. Logging and sampling information is entered into a computer using Excel software. Selected intervals of core are photographed however the entire drillhole is not systematically photographed.

After logging the core is prepared for sampling. A line is drawn down the core and the cutter uses this as a guide. The entire drillhole is then cut. A Columbus geologist does the actual sampling.

The core is sampled at 1 m intervals using the measuring blocks prepared upon initial receipt of the drill core as a guide. The entire drillhole is sampled at an average of 1 m intervals; sample lengths are adjusted to honor lithological contacts and mineralized intervals. Half of the drill core is placed in a plastic sample bag while the other half is retained din the core box for future reference. Saprolite material is cut with a knife and half placed in a textile bag for assay and the other half returned to the core box The samples and sample bags are numbered sequentially in advance allowing for the insertion standard reference samples, duplicates and blanks. The plastic sample bags are placed in larger rice bags and sealed for shipping. The sample bags are then sent by air transport to Cayenne and dropped off by SOTRAPMAG personnel to the Filab depot in Cayenne, followed by road transport from Cayenne to the laboratory in Paramaribo, Suriname for preparation and analyses.

All the core from Columbus's drilling is stored in covered core racks at the Citron exploration camp (Figure 11.2.1.2).

Source: Columbus, 2015

Figure 11.2.1.2: Core Racks at the Citron Exploration Camp

11.2.2 Density Measurements

Columbus measures the bulk density of representative samples of the various rock types and not the bulk density in each drillhole. They used a conventional bulk density scale with a basket suspended below the scale to allow immersion in water. Samples are not coated in paraffin wax, however, the core was observed to be generally solid with very little pores. Saprolite was wrapped in cellophane.

The following measurement methodology was employed:

- Weigh the sample to determine the dry mass;
- Place the sample in a basket and weigh it, suspended from a balance, in (under) water. Subtract the weight of the basket in (under) water, to determine the mass of the sample in water; and
- The relative dry bulk density, a unit-less ratio, is calculated as the dry mass of the sample in air divided by the difference in the mass of the sample in air and the mass of the sample in water.

The scale is zeroed out before each use and the weight of the basket holding the core is repeatedly measured.

Water density is assumed to be 1.0 t/m^3 with no adjustment made for changes in water temperature. Since all measurements were performed indoor in normal air temperatures, the actual water density should range between 0.999 t/m^3 at 15° C to 0.997 t/m^3 at 25°C. Therefore, assuming a value of 1.0 t/m^3 for water density will not introduce a significant bias in the estimate and is to industry standards.

As of November 2014, Columbus had made a total of 3,323 bulk density measurements on Montagne d'Or drill core. Bulk Density measurements were recorded for 9 different rock units (Table 11.2.2.1).

Table 11.2.2.1: Listing of Montagne d'Or Prospect Dry Rock Density Measurements

Rock Type	Number of Measurements	Average Density g/cm^3
Saprolite	354	1.695
Saprolite-Rock Transition	193	2.365
Felsic Tuff	1,056	2.911
Mafic Volcanics	413	3.154
Granodiorite	615	2.754
Feldspar Porphyry	61	2.786
Quartz-Feldspar Porphyry	164	2.817
Lapilli Tuff	75	2.864
Diabase Dikes	392	3.016

Source: SRK, 2015

11.2.3 Sample Preparation and Analysis

Columbus staff log and sample drill core but do not carry out any form of sample preparation (crushing/pulverizing) or analytical work on Project samples. All Project analytical work including sample preparation and analytical work is completed by FILAB at their laboratory in Paramaribo, Surinam.

FILAB established for several years a system of Quality Management and Safety to meet customer requirements (standards ISO17025 and ISO9001). FILAB is accredited by the DKD (now the DAkkS) and the SAFRAN Group and approved by DF control PMUC. The following description is sourced from documentation provided by FILAB.

After samples are received at the laboratory, then weighed and dried in furnaces at a temperature <130°C. They are then crushed and ground to a 70% <2.5 mm. From this grind a 300 to 400 g split is pulverized to 90% <100 µm. All equipment is cleaned by air after the processing of each sample.

Gold concentrations for the Columbus program are analyzed by FILAB using a 30 g sample split and fire assay pre-concentration methods followed by an atomic absorption spectroscopy finish (FA-AAS). The detection limit for this method is 0.01 ppm Au.

Gravimetric analysis was conducted on samples above a 5 g/t Au value for the 2013 and 2014 drilling program (the threshold is not reported for the earlier drilling and cannot be verified as personnel involved is no longer on site) and results from the gravimetric analysis were prioritized over FA in the database.

ICP analysis for up to 40 elements but routinely only for Ag and Cu are done using Aqua Regia digestion on a 0.25 g subsample.

FILAB routinely inserts blanks and certified reference materials (standards) into each batch of samples as an internal check.

11.3 Quality Assurance / Quality Control (QA/QC)

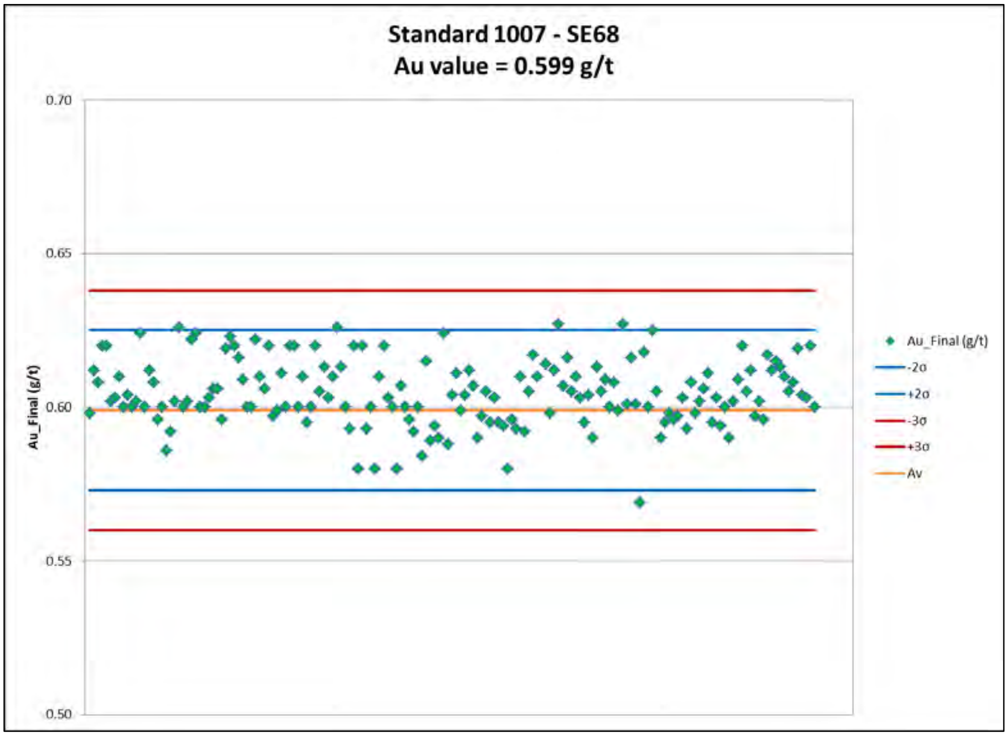
The QA/QC of all exploration data prior to June 29, 2014 has been presented in prior technical reports. The information presented below relates to the most recent exploration drilling conducted by Columbus during 2013 and 2014.

The Columbus QA/QC protocol of the 2013-2014 drilling programs included 14 different commercially certified standard reference materials for Au and blanks. The Columbus standards ranged between 0.599 to 8.981 g/t Au, which represents the typical levels of gold mineralization in the deposit. Standards are blindly inserted to the sample stream at a rate of 1:20 samples. The results of the standard analysis must be within ±2 standard deviations of the mean to pass the initial validation. In the case of standard result is between ±2 and ±3 standard deviations, a more complete check is made to determine if the result is valid or not. If the standard is outside a mineralized zone, reanalysis of the batch is not necessary. If two standards in succession, return results between ±2 and ±3 standard deviation, the batch is typically reanalyzed. If the standard value is outside ±3 standard deviations, the value is considered as erroneous and the entire batch is reanalyzed by the laboratory.

Columbus blanks are blindly inserted with at least one per batch with the blank located after an interpreted zone of mineralization. Blanks used during the program came from a granite quarry located near Cayenne. The blank analysis is considered valid if its value is lower than 5 times the limit of detection (0.005x5 = 0.025 ppm), confirming that no contamination occurred. If the analysis is beyond 5 times the limit of detection, the entire batch is reanalyzed by the laboratory.

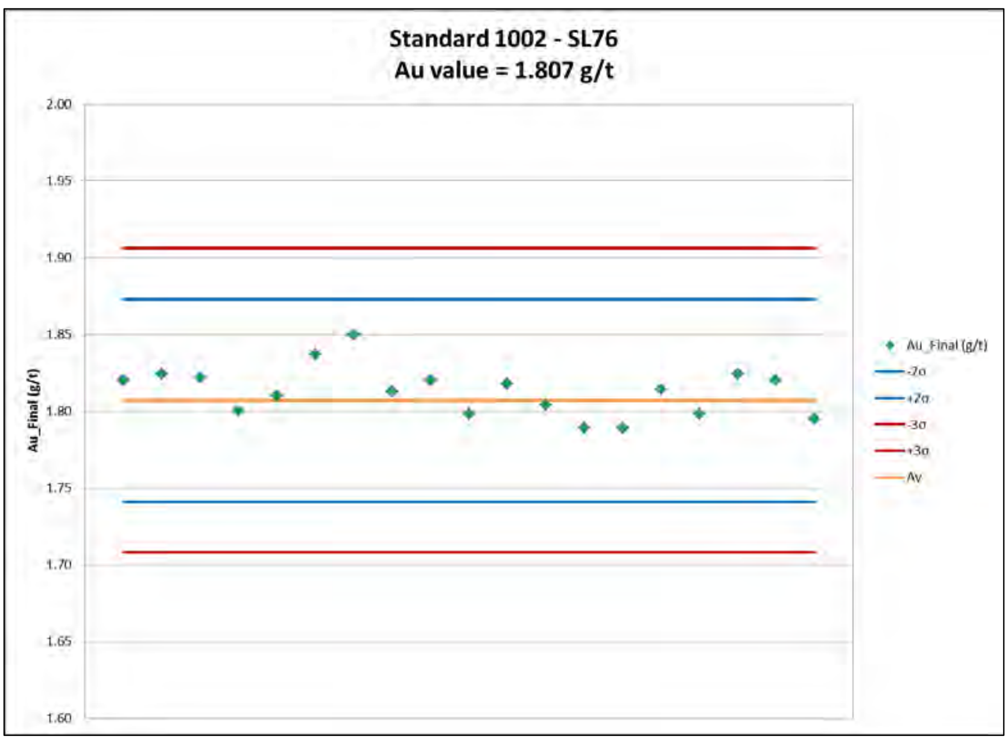
The laboratory conducts four types of internal QA/QC. They utilize two types of duplicates, standards and blanks. The laboratory uses duplicate pulps, generated and analyzed at a typical rate of 1:30 samples. Duplicate analyses of the same pulp are run at a typical rate of 1:15 samples.

QA/QC results are compiled in Excel as monthly reports. A representative set of standards at three typical grades and the blank results from the 2013-2014 drilling program are presented in Figures 11.3.1 to 11.3.4.



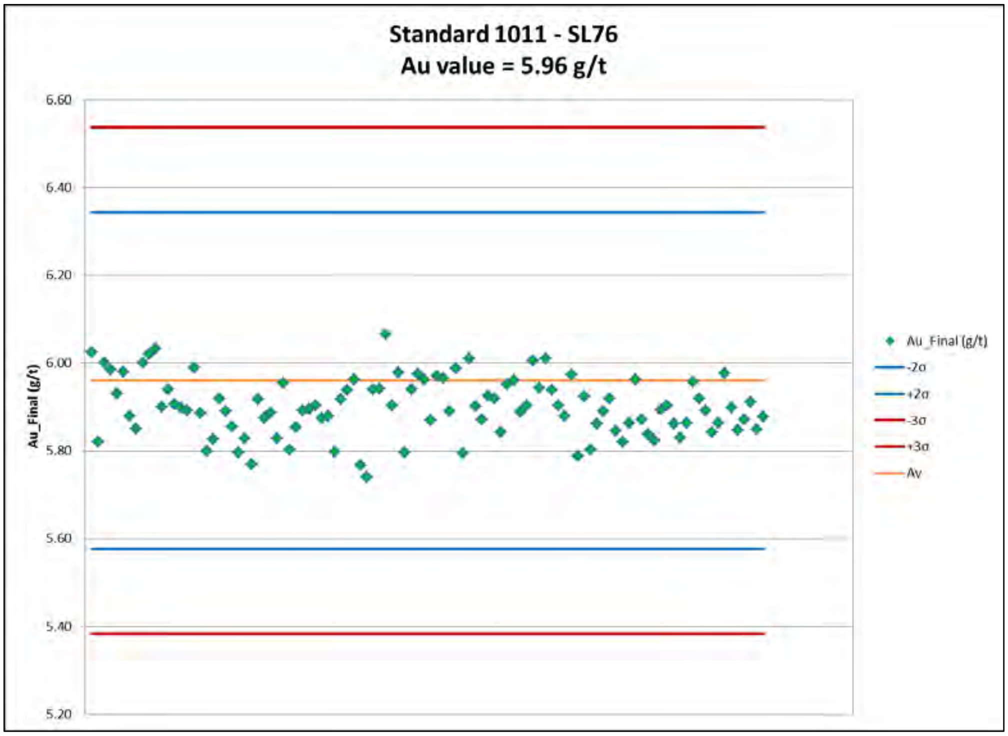
Source: Columbus, 2015

Figure 11.3.1: Results of Au Standard at 0.599 g/t



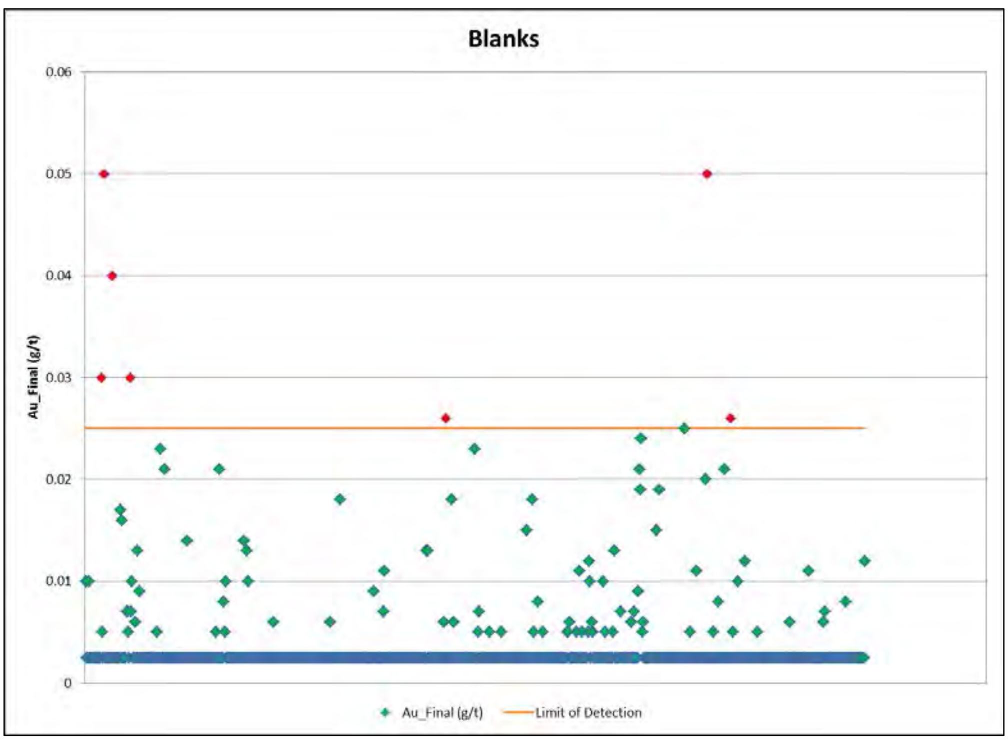
Source: Columbus, 2015

Figure 11.3.2: Results of Au Standard at 1.807 g/t



Source: Columbus, 2015

Figure 11.3.3: Results of Au Standard at 5.96 g/t



Source: Columbus, 2015

Figure 11.3.4: Results of all Blank Analyses

11.3.1 Conclusions

SRK is of the opinion that best professional judgment, and appropriate exploration and scientific methods were utilized in the preparation and analysis of the samples used in this report. SRK has reviewed the QA/QC results of the 2013-2014 drilling programs. SRK finds that the QA/QC program was well planned, executed and monitored. The standards are all certified and of appropriate levels of Au mineralization. The blank material is sufficiently hard so that it will scrub the sample preparation equipment to reveal any cross contamination. The results of the standards confirm there is no bias of the analytical lab. They also confirm that the laboratory has produced results with industry standard precision and accuracy. The blanks submitted with the QA/QC samples have shown that cross contamination or possible sample mix-ups are rare and do have a material impact on the analytical results.

12 Data Verification

12.1 Procedures

The database constructed prior to June 29, 2014 has been validated by previous QP's in order to support prior resource estimations. SRK validated the assay database by conducting systematic comparisons between the original assay certificate PDF copies to the electronic excel spreadsheet. Systematically spaced data was copied from a range of certificates that cover all of the new assays and was pasted directly into the Excel assay database for comparison. A total of 440 entries were checked, representing 2.5% of the new assay data. No discrepancies were found.

12.2 Limitations

SRK was not materially limited in its access to the supporting data used for the resource estimation. The database verification is limited to the procedures described above. All Mineral Resource data relies on the industry professionalism and integrity of those who collected and handled it. SRK is of the opinion that appropriate scientific methods and best professional judgment were utilized in the collection and interpretation of the data used in this report. However, users of this report are cautioned that the evaluation methods employed herein are subject to inherent uncertainties.

12.3 Opinion on Data Adequacy

It is SRK's opinion that the drillhole data is adequate to support the resource estimation of this report at the current level of resource classification. The database was constructed by Columbus under industry standard QA/QC protocols. Columbus maintains the database using GeoTic IOG an integrated database management system specifically designed to minimize the possibilities for data entry or data transfer errors. SRK's evaluation and subsequent validation of the database has provided good confidence in the data files.

13 Mineral Processing and Metallurgical Testing

Bureau Veritas Commodities Canada Ltd. - Inspectorate Metallurgical Division (Inspectorate) was retained by Nordgold to perform metallurgical testing on samples from the Project located in north-west French Guiana. The test program was directed and supervised by Eric Olin from SRK. The results of this metallurgical investigation are fully documented in Inspectorate's report, "Metallurgical Testing to Recover Gold and Silver from the Montagne d'Or Gold Project, French Guiana", March 30, 2015.

The test program was focused on the testing of two master composites formulated from available whole core intervals representing the Upper Felsic Zone (UFZ) and the Lower Favorable Zone (LFZ), as well as selected variability composites.

Three process options, including whole-ore cyanidation, a combination of gravity concentration followed by cyanidation of gravity tailing, and gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate, were investigated on two master composites, and the preferred process option and optimal conditions were further verified on ten variability test composites.

13.1 Sample Compositing

The metallurgical program was conducted on whole-core drillhole intervals derived from six metallurgical drillholes. The HQ size drillholes were planned based on the following criteria:

- Twinning of previous drillholes that intersected representative gold-copper intersections of variable grades across the principal felsic volcanic hosted UFZ and mixed volcanic hosted LFZ;
- A minimum of four intersections across UFZ and two across the LFZ, uniformly distributed along the east-west strike extent of the Montagne d'Or resources; and
- Intersections of the UFZ and LFZ in fresh rock below the weathered and oxidized saprolitic layer.

The drill core intervals selected for this metallurgical program are shown in Table 13.1.1 and the drillhole locations are shown in Figure 13.1.1.

Table 13.1.1: Drillholes and Intervals Used for the Metallurgical Program

Hole ID	Zone	From (m)	To (m)	Length (m)	Core Wt. (kg)
MET-14-01	UFZ	108.0	145.0	37	315
MET-14-02	UFZ	37.8	54.6	17	143
	UFZ	68.6	71.6	3	26
	UFZ	77.6	80.6	3	26
	UFZ	89.6	94.6	5	43
	LFZ	163.1	190.2	27	230
Total				**55**	**467**
MET-14-03	UFZ	37.0	163.0	126	1,071
MET-14-04	Sap	2.0	51.8	50	423
	UFZ	51.8	78.8	27	230
	LFZ	125.0	150.6	26	218
Total				**102**	**870**
MET-14-05	UFZ	71.0	109.0	38	323
	UFZ	130.0	136.0	6	51
Total				**44**	**374**
MET-14-06	LFZ	79.6	103.5	24	203
Total Core				**388**	**3,300**

Source: Inspectorate, 2015

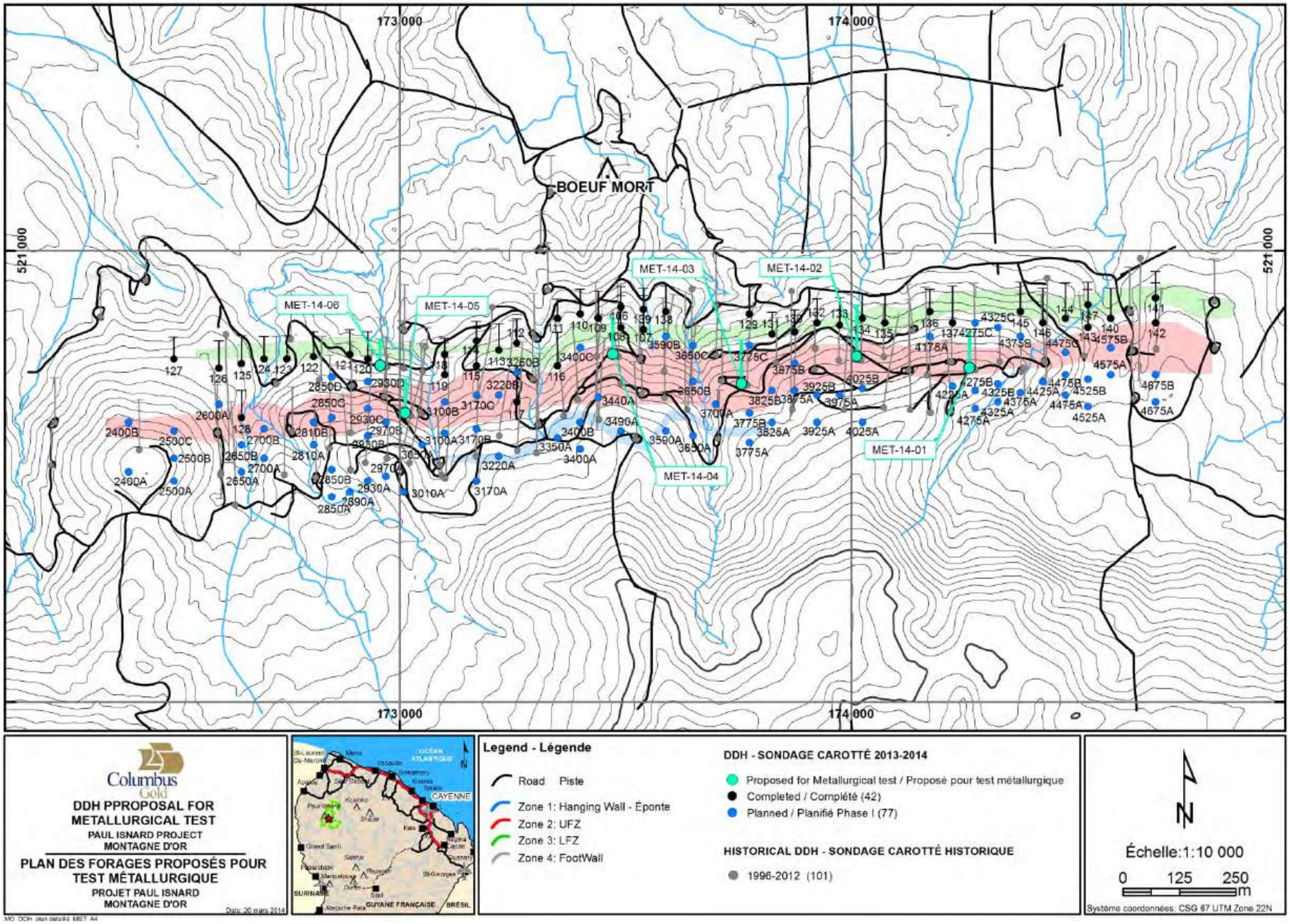


Figure 13.1.1: Metallurgical Drillhole Locations

Prior to sample preparation, whole core pieces from selected drillholes and intervals were handpicked for comminution testing, including semi-autogenous grinding (SAG) mill comminution (SMC), Abrasion index (Ai) and Bond ball mill work index (BWi) tests. Following the removal of the comminution samples, each of the HQ core intervals was stage-crushed separately to 100% passing 6-Tyler mesh. After riffle homogenization, a sub-sample was removed from each interval by rotary splitter and pulverized for head assays including Au, Ag, Cu, total S and sulfide S.

The intervals used to formulate the UFZ master composite and six UFZ variability composites are identified in Table 13.1.2 and Table 13.1.3, respectively. Table 13.1.4 and Table 13.1.5 identify the intervals used to formulate the LFZ master composite and three LFZ variability composites. As shown in Table 13.1.6, the interval from 2 to 51.8 m in drillhole MET14-04 was used for formulating the saprolite variability composite. The variability composites were intended represent spatial variability as well as variability with respect to gold and copper grades:

- UFZ – VC1: medium gold and low copper
- UFZ – VC2: low gold and low copper
- UFZ – VC3: medium gold and low copper
- UFZ – VC4: high gold and medium copper
- UFZ – VC5: medium gold and low copper
- UFZ – VC6: high gold and high copper
- LFZ – VC1: high gold and high copper
- LFZ – VC2: low gold and medium copper
- LFZ – VC3: medium gold and low copper

Table 13.1.2: UFZ Master Compositing List

Hole ID	From (m)	To (m)	Length (m)	Weight (kg)
MET-14-01	108.0	145.0	37.0	79
MET-14-02	37.8	54.6	16.8	36
	68.6	71.6	3.0	6
	77.6	80.6	3.0	6
	89.6	94.6	5.0	11
MET-14-03	37.0	163.0	126.0	268
MET-14-04	51.8	78.8	27.0	57
MET-14-05	71.0	109.0	38.0	81
	130.0	136.0	6.0	13
Total				**556**

Source: Inspectorate, 2015

Table 13.1.3: UFZ Variability Compositing List

UFZ Variability Composite ID	Interval				Weight (kg)
	Hole ID	From (m)	To (m)	Length (m)	
UFZ-VC1 (Medium Au and Low Cu)	MET-14-01	108.0	145.0	37.0	79
UFZ-VC2 (Low Au and Low Cu)	MET-14-02	37.8	54.6	16.8	36
		68.6	71.6	3.0	6
UFZ-VC3 (High Gold and Medium Cu)	MET-14-02	77.6	80.6	3.0	6
		89.6	94.6	5.0	11
UFZ-VC4 (Low Gold and Low Cu)	MET-14-03	37.0	163.0	126.0	268
UFZ-VC5 (Medium Gold and Low Cu)	MET-14-04	51.8	78.8	27.0	57
UFZ-VC6 (High Gold and High Cu)	MET-14-05	71.0	109.0	38.0	81
		130.0	136.0	6.0	13

Source: Inspectorate, 2015

Table 13.1.4: LFZ Master Compositing List

Hole ID	From (m)	To (m)	Length (m)	Weight (kg)
MET-14-02	163.1	190.2	27.1	86
MET-14-04	125.0	150.6	25.6	82
MET-14-06	79.6	105.3	25.7	76
Total				**244**

Source: Inspectorate, 2015

Table 13.1.5: LFZ Variability Compositing List

LFZ Variability Composite ID	Interval				Weight
	Hole ID	From (m)	To (m)	Length (m)	(kg)
LFZ-VC1 (High Au and High Cu)	Met-14-02	163.1	190.2	27.1	86
LFZ-VC2 (Low Au and Medium Cu)	Met-14-04	125.0	150.6	25.6	82
LFZ-VC3 (Medium Au and Low Cu)	Met-14-06	79.6	105.3	23.9	76

Source: Inspectorate, 2015

Table 13.1.6: Saprolite Variability Compositing List

Hole ID	From (m)	To (m)	Length (m)	Weight (kg)
Met-14-04	2.0	51.8	49.8	106

Source: Inspectorate, 2015

13.2 Test Composite Characterization

13.2.1 Chemical Analyses

The gold and silver assays were conducted on each composite by FA in triplicate and by metallic screen procedures. The UFZ master composite averaged about 1.54 g/t Au and 3.1 g/t Ag. The LFZ master composite averaged 1.54 g/t Au and 5.0 g/t Ag. The variability composites ranged from 0.84 to 3.65 g/t Au and 1.6 to 9.0 g/t Ag. The gold and silver assays are presented in Table 13.2.1.1. Additionally, all master and variability composites were analyzed for cyanide soluble gold, sequential copper, mercury, sulfur and carbon speciation as well as ICP metals. The main assays of interest are presented in Table 13.2.1.2. The average copper content in the test composites was 0.1% Cu, which was generally present as primary copper. The presence of acid and cyanide soluble copper was relatively low. The total sulfur content varied from 0.7 % to 4.9% and was primarily present as sulfide sulfur. In general, the LFZ master composite contained higher sulfur than the UFZ master composite. The carbon contents were very low, indicating that preg-robbing will likely not occur during cyanidation. Mercury ranged from 0.04 to 0.35 ppm in the master composites and 0.01 to 1.91 ppm in the variability composites.

Table 13.2.1.1: Gold and Silver Analyses on UFZ and LFZ Master and Variability Composites

Composite ID	By Direct FA in Triplicate						By Metallic		Average	
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
UFZ Master Comp.	0.84	3.0	2.24	4.0	1.41	3.0	1.66	2.5	1.54	3.1
UFZ-VC1	1.52	2.0	1.27	2.0	1.32	1.0	4.17	1.8	2.07	1.7
UFZ-VC2	0.76	2.0	1.45	2.0	0.54	1.0	0.62	1.3	0.84	1.6
UFZ-VC3	1.34	2.0	1.25	2.0	2.59	<1	4.25	1.1	2.36	1.7
UFZ-VC4	1.21	3.0	0.84	2.0	1.86	3.0	0.96	1.0	1.22	2.3
UFZ-VC5	0.76	2.0	1.09	2.0	0.87	2.0	1.00	2.0	0.93	2.0
UFZ-VC6	2.21	5.0	4.31	7.0	2.37	6.0	2.57	5.0	2.87	5.8
UFZ Zone Average									**1.69**	**2.6**
LFZ Master Comp.	1.35	6.0	1.82	4.0	1.51	4.0	1.50	5.8	1.55	5.0
LFZ-VC1	2.31	10.0	2.26	8.0	7.45	11.0	2.58	7.1	3.65	9.0
LFZ-VC2	0.46	4.0	0.49	1.0	3.04	2.0	1.15	3.5	1.29	2.6
LFZ-VC3	0.71	4.0	1.15	5.0	0.98	4.0	1.21	6.5	1.01	4.9
LFZ Zone Average									**1.87**	**5.4**
Saprolite Var. Comp.	1.62	2.0	0.52	1.0	0.66	1.0	1.05	1.3	0.96	1.3

Source: Inspectorate, 2015

Table 13.2.1.2: Elemental Analyses on UFZ and LFZ Master and Variability Composites

Items	Units	UFZ Zone						
		Master Comp.	UFZ-VC1	UFZ-VC2	UFZ-VC3	UFZ-VC4	UFZ-VC5	UFZ-VC6
Au	g/t	1.54	2.07	0.84	2.36	1.22	0.93	2.87
Ag	ppm	3	2	2	2	2	2	6
Au (CN Soluble)	g/t	0.74	1.05	0.40	0.91	0.63	0.75	1.02
Cu	%	0.10	0.07	0.07	0.10	0.09	0.02	0.21
Cu(A.S.)	%	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Cu(CN)	%	<0.01	<0.01	0.01	<0.01	0.01	0.01	0.01
Cu(Resid.)	%	0.08	0.05	0.04	0.08	0.06	0.01	0.18
S(ele)	%	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
S(-2)	%	1.67	2.39	0.67	1.89	1.61	0.52	2.66
S(tot)	%	1.70	2.41	0.71	1.91	1.63	0.68	2.68
S(SO_4)	%	0.03	0.02	0.04	0.02	0.02	0.16	0.02
C(tot)	%	0.04	0.10	0.02	<0.02	0.04	<0.02	0.04
C(Org)	%	0.04	0.06	0.02	<0.02	0.04	<0.02	0.04
C Graphite	%	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
C(Inorg)	%	<0.02	0.04	<0.02	<0.02	<0.02	<0.02	<0.02
Hg	ppm	0.35	0.03	0.06	0.08	0.09	0.07	1.91

Items	Units	LFZ Zone				Saprolite Var.
		Master Comp.	LFZ-VC1	LFZ-VC2	LFZ-VC3	
Au	g/t	1.55	3.65	1.29	1.01	0.96
Ag	ppm	5	9	3	5	1
Au (CN Soluble)	g/t	0.81	1.15	0.33	0.59	0.90
Cu	%	0.13	0.29	0.02	0.06	0.04
Cu(A.S.)	%	<0.01	<0.01	<0.01	<0.01	<0.01
Cu(CN)	%	0.01	0.02	<0.01	<0.01	<0.01
Cu(Resid.)	%	0.09	0.22	0.01	0.04	0.03
S(ele)	%	<0.02	<0.02	<0.02	<0.02	<0.02
S(-2)	%	2.45	4.94	0.80	1.35	<0.02
S(tot)	%	2.47	4.97	0.82	1.37	0.02
S(SO_4)	%	0.02	0.03	0.02	0.02	0.02
C(tot)	%	0.06	0.02	0.04	0.11	0.06
C(Org)	%	0.04	0.02	0.03	0.07	0.06
C Graphite	%	<0.02	<0.02	<0.02	<0.02	<0.02
C(Inorg)	%	0.02	<0.02	<0.02	0.04	<0.02
Hg	ppm	0.04	0.06	0.12	0.01	0.01

Source: Inspectorate, 2015

13.2.2 Mineralogical Analyses

Representative sub-samples of the UFZ and LFZ master composites were examined by Quantitative Evaluation of Minerals by Scanning Electron (QEMSCAN) to identify the types of minerals and bulk associations, and to provide quantitative information on mineral percentages, particle size, shape, degree of liberation and locking analysis, and carrier mineral inspections for gold and silver. The results of these mineralogical analyses are fully documented in Inspectorate's report, "Mineralogical Study on the Master Composites," January 2015.

Each composite was ground to a P_{80} of 75 µm and then screened into six sized fractions, varying from 105 to 25 µm, for automated mineral analysis. Polished block sections were prepared from each fraction and then systematically scanned using QEMSCAN. Due to the relatively low grade of gold and silver in the test composites, pre-concentration using a Knelson concentrator was performed on ~6 kg of each master composite to produce rougher gravity concentrate for gold and silver deportment mineralogy studies using the QEMSCAN Trace Mineral Search (TMS). Table 13.2.2.1 shows the percentage mineral composition in each of the master composites. Key findings from the mineralogical study were:

- The main sulfide minerals in the two master composite were pyrite and pyrrhotite, which accounted for 3.1% to 3.8% of the total mass. Chalcopyrite was the principal copper bearing mineral, and carried 98% of the copper in the test samples. Only trace amounts of copper were contained in chalcocite/covellite, bornite, and tetrahedrite. Other sulfide minerals, including sphalerite, galena, arsenopyrite, bismuthnite, cobaltite, and FeNi(Co)-sulfarsenide, were all at trace levels.
- The sulfide minerals were contained in a silicon rich non-sulfide gangue host. Over 95% of the non-sulfide minerals occurred as different types of silicates: including quartz, feldspar group minerals, muscovite/illite/biotite, chlorite, amphibole/pyroxene and kaolinite. The iron oxides occurred mostly as magnetite, hematite and ilmenite.
- The majority of the gold grains in the test gravity concentrate were present as native gold or gold electrum sized < 20 µm (12 to 13 µm on average). However, the coarsely grained gold, sized >30 µm, carried about 90% of the gold contained in the gravity concentrates. In comparison to LFZ composite, UFZ composite contained relatively higher amounts of native gold. In addition, the gold-mercury bearing mineral, goldamalgam [(Au,Ag)Hg], was observed in the UFZ composite.
- The gold liberation data showed that less than a quarter of the gold in the test composite was liberated. The unliberated gold was mostly interlocked with pyrite and non-sulfide gangue. A relatively low amount of gold was associated with chalcopyrite and sphalerite.
- Most of the silver (>90%) was contained in gold or gold minerals. The other silver minerals noticed including native silver/eugenite, freibergite, ourayite, hessite, acanthite/argentite, stephanite and matildite.

Table 13.2.2.1: Mineral Percentages in UFZ and LFZ Master Composites

Mineral Contents (wt. %)					
Sulfide Minerals	**LFZ**	**UFZ**	**Non-sulfide Minerals**	**LFZ**	**UFZ**
Copper Sulfides	0.32	0.27	Jarosite	0.00	0.05
Pyrite	2.57	2.55	Iron Oxides	1.25	0.98
Pyrrhotite	1.24	0.54	Quartz	37.7	39.9
Sphalerite	0.03	0.01	Feldspars	16.5	12.7
Galena	0.01	0.00	Micas	26.7	29.8
			Chlorite	4.63	5.1
			Other Silicates	7.68	7.0
			Others	1.30	1.10
Total	**4.18**	**3.38**		**95.8**	**96.6**

Source: Inspectorate, 2015

13.2.3 Hardness Characterization

BWi, Bond Ai and SMC tests were conducted on split core samples representing the UFZ and LFZ master test composites, while only the BWi was conducted on the ten variability composite samples. The BWi and Ai test results are summarized in Table 13.2.3.1. The SMC results are presented in Table 13.2.3.2. The BWi for the UFZ and LFZ master composites was similar with reported results of 11.0 and 11.6 kWh/t, respectively. The BWi for the variability composites ranged from 9.0 to 12.8

kWh/t. These results indicate that material from the Montagne d'Or resource is of moderate hardness.

Table 13.2.3.1: Bond Ball Mill, Crushing and Abrasion Index Results

Zone	Comp. ID	Bond Crusher Impact Work Index (kWh/t)	Bond Abrasion	Bwi (kWh/t)
UFZ	UFZ VC1 Comp.			12.3
	UFZ VC2 Comp.			10.1
	UFZ VC3 Comp.			9.0
	UFZ VC4 Comp.			10.2
	UFZ VC5 Comp.			9.7
	UFZ VC6 Comp.			12.8
	Average of Variability Comp.			**10.7**
	UFZ Master Comp.	**9.2**	**0.1061**	**11.0**
LFZ	LFZ VC1 Comp.			11.6
	LFZ VC2 Comp.			11.6
	LFZ VC3 Comp.			12.1
	Average of Variability Comp.			**11.8**
	LFZ Master Comp.	**8.6**	**0.0933**	**11.6**
Sap.	Saprolite Variability Comp.			6.8

Source: Inspectorate, 2015

Table 13.2.3.2: Summary of SMC Evaluation

Sample ID	SG	A	b	A x b	DW_i kWh/m^3	DW_i (%)	M_{ia} (kWh/Mt	M_{ih} (kWh/Mt)	M_{ic} (kWh/Mt)	t_a
Zone LFZ Master Comp.	2.85	57.7	0.64	36.9	7.66	75	20.6	15.6	8.1	0.34
Zone UFZ Master Comp.	2.94	58.5	0.57	33.3	8.75	84	22.1	17.3	8.9	0.30

Source: Inspectorate, 2015
SMC Parameters:
A = maximum breakage
b = relation between energy and impact breakage A × b = overall AG–SAG hardness
DW_i = drop-weight index
M_{ia} = coarse particle component
M_{ih} = high-pressure grinding roll component Mic = crusher component
t_a = low energy abrasion component of breakage

13.3 Metallurgical Testwork – Master Composites

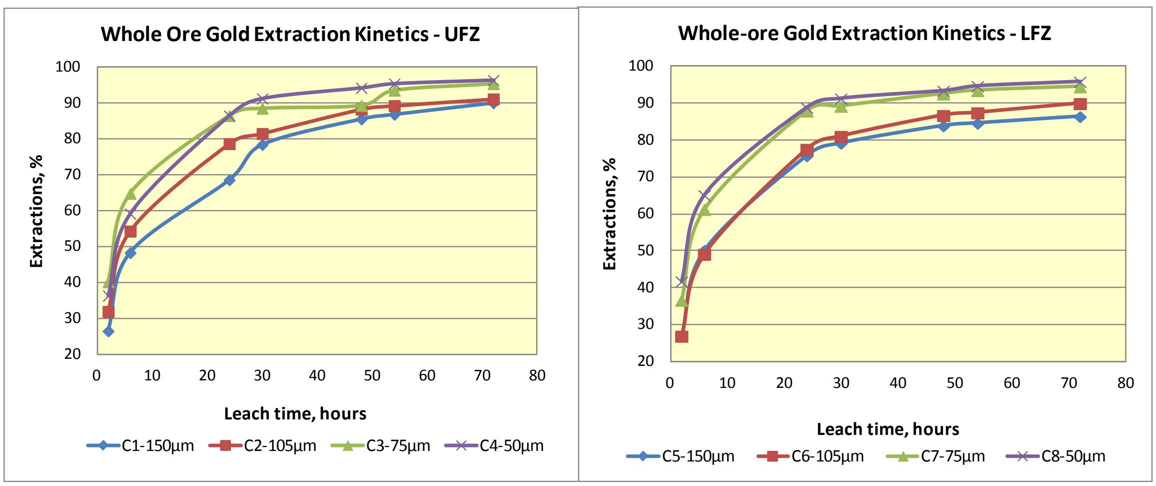
13.3.1 Whole-Ore Cyanidation

Whole-ore bottle roll cyanidation tests were conducted on the UFZ and LFZ master composites at sizes of 80% passing (P_{80}) of 150, 105, 75 and 50 µm to assess the effect of grind size on gold extraction, leach kinetics and reagent requirements. The leach tests were carried out at 40% solids for 72 hours at a cyanide concentration of 1.0 g/L NaCN with pH maintained at 10.5 to 11 with hydrated lime. The results of these tests are summarized in Table 13.3.1.1, and leach kinetics are presented in Figure 13.3.1.1. Gold extractions from the UFZ master composite ranged from 90.1% to 96.4% as the grind size became progressively finer. Gold extractions from the LFZ master composite ranged from 86.4% to 95.8% over the range of grind sizes tested. These test results indicate that the optimum grind size is about P_{80} 75 µm.

Table 13.3.1.1: Summary of Whole-Ore Cyanidation versus Grind Size

Composite ID	Test No	P80, µm	Measured Head Au (g/t)	Calculated Head Au (g/t)	Gold Recovery Au (%)	Residue Grade Au (g/t)	Consumption (kg/t)	
							NaCN	Ca(OH)2
UFZ Master Comp.	C1	149	1.54	1.82	90.1	0.18	1.58	0.25
	C2	102	1.54	1.70	91.2	0.15	1.53	0.20
	C3	77	1.54	1.42	95.4	0.07	1.56	0.20
	C4	52	1.54	1.94	96.4	0.07	1.65	0.22
LFZ Master Comp.	C5	151	1.55	1.84	86.4	0.25	1.74	0.13
	C6	107	1.55	2.18	89.9	0.22	1.77	0.15
	C7	75	1.55	2.17	94.5	0.12	1.77	0.14
	C8	52	1.55	2.88	95.8	0.12	2.00	0.13

Source: Inspectorate, 2015



Source: Inspectorate, 2015

Figure 13.3.1.1: Gold Extraction versus Leach Retention Time

13.3.2 Gravity Concentration + Cyanidation of Gravity Tailing

Gravity Concentration + Cyanidation Versus Grind Size

As an alternative process route to whole-ore cyanidation, a combination of gravity pre-concentration followed by cyanide leaching of gravity tails was investigated on the UFZ and LFZ master composites at grind sizes of P_{80} 150, 105, 75 and 50 µm. The results of these tests are summarized in Table 13.3.2.1. Ground samples were subjected to single-pass gravity concentration with a Knelson centrifugal separator (Model KC-MD3). The Knelson rougher gravity concentrate was then hand-panned to simulate cleaning. The entire cleaner concentrate was fire assayed for gold. Combined pan tails and gravity tails were re-pulped to 40% solids and subjected to cyanide leaching using the same conditions as in whole-ore cyanidation tests.

Table 13.3.2.1: Summary of Gravity + Cyanidation Tests Versus Grind Size

Composite ID	Test No	P80, µm	Calculated Head Au (g/t)	Gold Recovery			Residue Grade Au (g/t)	Consumption (kg/t)	
				Gravity Au (%)	Cyanidation Au (%)	Overall Au (%)		NaCN	Ca(OH)2
UFZ Master Comp.	GC1	151	1.68	18.3	71.0	89.3	0.18	1.53	0.18
	GC2	102	1.72	25.3	65.4	90.7	0.16	1.66	0.18
	GC3	76	2.47	39.4	58.2	97.6	0.06	1.44	0.17
	GC4	52	1.77	32.2	65.5	97.7	0.04	1.71	0.20
LFZ Master Comp.	GC5	148	1.65	13.8	74.6	88.5	0.19	1.77	0.15
	GC6	102	1.61	17.7	77.3	95.0	0.08	1.78	0.15
	GC7	73	1.80	28.1	67.5	95.6	0.08	1.94	0.15
	GC8	49	1.72	28.3	69.4	97.7	0.04	2.04	0.15

Source: Inspectorate, 2015

Results showed that both master composites were highly amenable to gravity separation, with up to 39.4% gold recovery from the UFZ composite and up to 28.3% gold recovery from the LFZ master composite into the gravity cleaner concentrate. The results of whole-ore cyanidation and gravity + cyanidation are compared in Table 13.3.2.2 where it can be seen that gravity + cyanidation led to slightly better gold recovery and lower residual gold grades at the same grind.

Table 13.3.2.2: Comparison of Whole-ore Cyanidation and Gravity + Cyanidation Results

Composite	Target P80 Size (µm)	Gold Recovery (% Au)		Residual Grade (g/t Au)	
		Whole-ore Cyanidation	Gravity + Cyanidation	Whole-ore Cyanidation	Gravity + Cyanidation
UFZ Master Composite	150	90.1	89.3	0.18	0.18
	100	91.2	90.7	0.15	0.16
	75	95.4	97.6	0.07	0.06
	50	96.4	97.7	0.07	0.04
LFZ Master Composite	150	86.4	88.5	0.25	0.19
	100	89.9	95.0	0.22	0.08
	75	94.5	95.6	0.12	0.08
	50	95.8	97.7	0.12	0.04

Source: Inspectorate, 2015

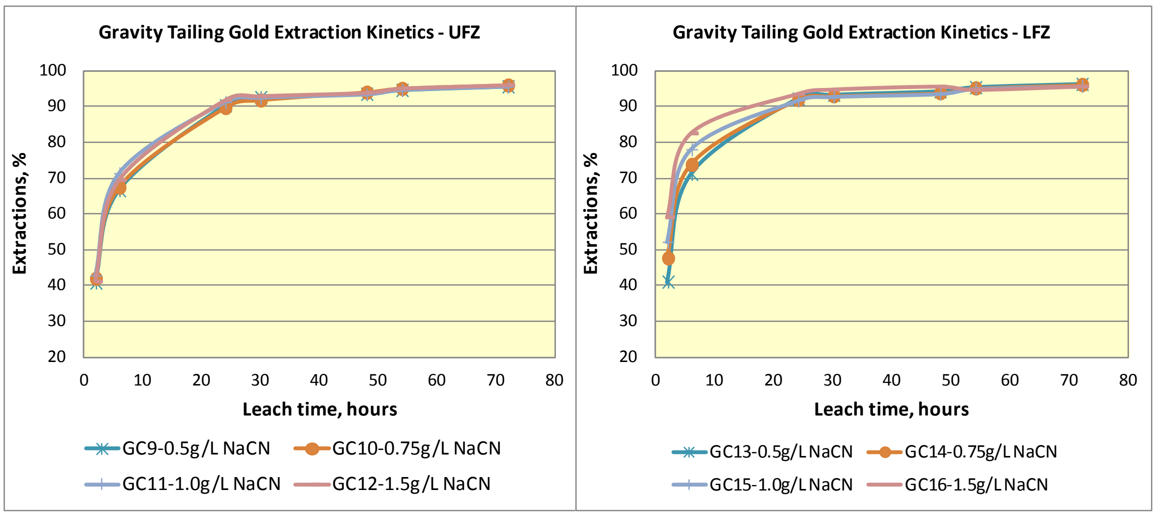
<u>Gravity Concentration + Cyanidation of Gravity Tailings Versus Cyanide Concentration</u>

Gravity concentration + cyanidation of the gravity tailings versus cyanide concentration were evaluated on both master composites at the optimum primary grind of P_{80} 75 µm. The cyanide concentration tests on each gravity tailing sample were carried out at 40% solids at cyanide concentrations of 0.5, 0.75, 1.0 and 1.5 g/L NaCN. The test results are summarized in Table 13.3.2.3. Increasing cyanide strength from 0.5 to 1.5 g/L resulted in negligible gains in gold recovery, with 96% to 97% gold recovery obtained regardless of the cyanide concentration; however, cyanide consumption was substantially reduced at the lower cyanide concentrations. Leach kinetics shown in Figure 13.3.2.1 indicate that a 48-hour residence time is sufficient for leaching gravity tailings.

Table 13.3.2.3: Cyanidation of Gravity Tailings versus Cyanide Concentration

Composite ID	Test No	NaCN (g/L)	Calculated Head Au (g/t)	Gold Recovery			Residue Grade Au (g/t)	Consumption (kg/t)	
				Gravity Au (%)	Cyanidation Au (%)	Overall Au (%)		NaCN	Ca(OH)2
UFZ Master Comp.	GC9	0.50	1.79	25.9	70.8	96.7	0.060	0.78	0.22
	GC10	0.75	1.79	26.2	70.7	96.9	0.055	1.17	0.20
	GC11	1.00	1.78	26.2	70.4	96.6	0.060	1.28	0.20
	GC12	1.50	1.74	27.4	69.7	97.1	0.050	1.53	0.20
LFZ Master Comp.	GC13	0.50	1.79	27.5	69.7	97.2	0.050	0.92	0.16
	GC14	0.75	1.86	25.9	71.2	97.0	0.055	1.43	0.16
	GC15	1.00	1.82	26.0	70.8	96.7	0.060	1.66	0.15
	GC16	1.50	1.82	25.8	70.9	96.7	0.060	2.23	0.15

Source: Inspectorate, 2015



Source: Inspectorate, 2015

Figure 13.3.2.1: Gold Extraction from Gravity Tailings versus Retention Time

13.3.3 Gravity Concentration + Flotation

As an alternative process, gravity concentration followed by gold flotation from the gravity tailing was investigated. These studies investigated different reagent regimes and grind sizes. The preferred reagent regime from the reagent evaluation tests was further tested on each master composite at three different grind sizes, ranging from 50 to 150 µm, to evaluate the primary grind requirement.

Results showed that both composites are highly amendable to the gravity + flotation process. The gravity circuit was able to remove 28% to 48% coarse gold into a gravity concentrate representing ≤0.01% mass and assaying up to 1 kg/t Au.

The grind-recovery test results for the UFZ and LFZ master composites are summarized in Tables 13.3.3.1 and 13.3.3.2, respectively. Results showed that both master composite samples were grind-sensitive. Gold recoveries of about 96% into gravity + rougher flotation concentrates were obtained at grinds less than P_{80} 105 µm. After a review of the grind-recovery test results on the three master composites, a target P_{80} grind of 75 µm was selected as an optimum grind. The presence of coarse free gold led to some variation in the calculated head grades on all samples tested.

Table 13.3.3.1: Gravity + Flotation Tests versus Grind Size on UFZ Master Composite

Summary for Gold																	
Test No	P80 Size (µm)	Gold Grade (g/t Au)								Gold Recovery (%)						Overall Recovery	
		Meas. Head	Calc. Head	Gravity Conc.	Flotation					Gravity Conc.	Flotation					Mass (%)	Au (%)
					Ro Conc. 1	Ro Conc.1-2	Ro Conc.1-3	Total Conc.	Tails		Ro Conc. 1	Ro Conc.1-2	Ro Conc.1-3	Total Conc.			
GF11	51	1.54	1.32	524.0	11.9	8.0	6.1	5.1	0.07	31.9	60.5	62.6	63.3	63.7		**16.7**	**95.6**
GF2	73	1.54	2.42	1,018.6	16.2	11.9	9.9	8.6	0.09	47.9	47.6	48.3	48.6	48.8		**16.1**	**96.8**
GF10	104	1.54	2.47	1,128.6	24.5	15.9	13.4	10.8	0.11	40.3	53.3	55.0	55.5	55.9		**12.9**	**96.1**
GF9	149	1.54	1.73	386.2	21.7	14.7	12.3	10.8	0.13	20.6	69.7	71.6	72.5	72.8		**11.8**	**93.4**
Summary for Silver																	
Test No	P80 Size (µm)	Silver Grade (g/t Au)								Silver Recovery (%)						Overall Recovery	
		Meas. Head	Calc. Head	Gravity Conc.	Flotation					Gravity Conc.	Flotation					Mass (%)	Ag (%)
					Ro Conc. 1	Ro Conc.1-2	Ro Conc.1-3	Total Conc.	Tails		Ro Conc. 1	Ro Conc.1-2	Ro Conc.1-3	Total Conc.			
GF11	51	3.13	2.56	250.0	27.0	18.2	14.0	11.7	0.50	n/a	71.1	74.0	75.3	75.9		**16.7**	**75.9**
GF2	73	3.13	2.44	<50	26.0	19.6	16.6	14.6	0.50	n/a	75.8	79.1	80.8	82.3		**16.1**	**82.3**
GF10	104	3.13	3.04	347.0	39.0	26.5	22.4	18.0	0.50	n/a	68.9	74.1	75.2	75.7		**12.9**	**75.7**
GF9	149	3.13	2.64	139.0	34.0	23.8	19.9	17.7	0.50	n/a	71.6	76.0	77.4	78.5		**11.8**	**78.5**

Source: Inspectorate, 2015

Table 13.3.3.2: Gravity + Flotation Tests versus Grind Size on the LFZ Master Composite

Summary for Gold																	
Test No	P80 Size (µm)	Gold Grade (g/t Au)									Gold Recovery (%)					Overall Recovery	
		Meas. Head	Calc. Head	Gravity Conc.	Flotation						Gravity Conc.	Flotation				Mass (%)	Au (%)
					Ro Conc. 1	Ro Conc.1-2	Ro Conc.1-3	Total Conc.	Tails			Ro Conc. 1	Ro Conc.1-2	Ro Conc.1-3	Total Conc.		
GF14	52	1.55	1.90	803.9	13.2	9.1	7.3	6.4	0.06		40.2	55.9	56.7	57.0	57.2	**16.9**	**97.4**
GF6	73	1.55	2.03	529.0	18.0	13.3	11.0	9.9	0.08		30.0	64.5	65.9	66.4	66.7	**16.0**	**96.6**
GF13	108	1.55	2.24	463.5	22.6	16.7	13.5	11.0	0.10		20.3	74.0	75.1	75.6	76.0	**15.6**	**96.2**
GF12	155	1.55	1.41	361.5	14.2	10.4	8.7	7.7	0.10		18.7	72.1	74.0	74.7	75.2	**13.8**	**93.9**
Summary for Silver																	
Test No	P80 Size (µm)	Silver Grade (g/t Au)									Silver Recovery (%)					Overall Recovery	
		Meas. Head	Calc. Head	Gravity Conc.	Flotation						Gravity Conc.	Flotation				Mass (%)	Ag (%)
					Ro Conc. 1	Ro Conc.1-2	Ro Conc.1-3	Total Conc.	Tails			Ro Conc. 1	Ro Conc.1-2	Ro Conc.1-3	Total Conc.		
GF14	52	4.95	5.50	631.0	53.0	37.1	30.1	26.6	0.50		10.9	77.7	79.7	80.8	81.5	**16.9**	**92.4**
GF6	73	4.95	5.46	233.0	62.0	46.5	38.5	34.8	0.50		4.9	82.8	85.9	86.7	87.2	**16.0**	**92.1**
GF13	108	4.95	5.41	408.0	59.0	44.4	36.1	29.5	0.50		7.4	80.3	82.8	83.7	84.8	**15.6**	**92.2**
GF12	155	4.95	5.71	161.0	63.0	46.6	39.1	34.6	1.00		2.0	79.1	81.6	82.3	82.9	**13.8**	**84.9**

Source: Inspectorate, 2015

Large Batch Gravity + Rougher Flotation Testwork

Large batch tests were conducted on each master composite to produce sufficient rougher products for regrind-cleaner flotation test work and downstream concentrate leach and cyanide detoxification studies. The large-batch tests were performed by grinding 20 kg test charges in a large rod mill to a target grind size of P_{80} 75 µm. Each ground batch was subjected to single-pass gravity concentration with a Knelson centrifugal gravity concentrator followed by panning of the primary gravity concentrate. The combined primary gravity and pan tails were then subjected to 4-stage rougher flotation in a 56L Denver Dual flotation cell. Rougher flotation was performed at natural pH and a total of 50 g/t PAX and 30 g/t A208 were added in four stages as mineral collectors.

The results of the large-batch tests on the UFZ master composites are summarized in Table 13.3.3.3. Gravity concentration resulted in an average gold recovery of 35% and an average silver recovery of 13.8% into a gravity cleaner concentrate that averaged 4.7 kg/t Au and 3.2 kg/t Ag. Flotation of the gravity tails recovered an additional 60.4% of gold and 70.8% of silver, into a rougher flotation concentrate that averaged 6.52 g/t Au and 13.3 g/t Ag, at approximately 14.7% of the original mass. Combined gravity plus flotation gold and silver recoveries averaged 95.4% and 84.4%, respectively.

The results of the large-scale batch tests on the LFZ master composite are summarized in Table 13.3.3.4. Gravity concentration recovered an average of 32.1% of gold and 9.3% of silver into a gravity concentrate containing 4.1 kg/t Au and 3.8 kg/t Ag representing 0.01% of the original feed mass. Flotation of gravity tails recovered an additional 64.5% of gold and 84.6% of silver into rougher flotation concentrates that averaged 7.19 g/t Au and 31 g/t Ag, resulting in a combined gravity + flotation gold recovery of 96.6% and a combined silver recovery of 93.8%.

Table 13.3.3.3: Summary of Large Batch Gravity + Flotation Tests Results on the UFZ Master Composite

Balance for Gold										
Test No.	P80 Size (µm)	Assay			Mass (%)	Recovery			Calculated Head	Measured Head
		Pan Conc. Au (g/t)	Flotation Conc. Au (g/t)	Flotation Tailing Au (g/t)		Gravity % Au	Flotation % Au	Overall % Au	Au (g/t)	Au (g/t)
BGF1	79	4786.5	6.14	0.08	14.2	37.0	58.4	95.4	1.49	1.54
BGF2	77	4180.0	6.63	0.08	14.3	32.3	63.1	95.4	1.50	1.54
BGF3	76	4945.4	6.37	0.10	16.7	32.9	62.3	95.1	1.71	1.54
BGF4	78	4995.2	6.92	0.08	13.5	38.0	57.7	95.7	1.62	1.54
Average	**78**	**4726.8**	**6.52**	**0.09**	**14.7**	**35.0**	**60.4**	**95.4**	**1.58**	**1.54**
Balance for Silver										
Test No.	P80 Size (µm)	Assay			Mass %	Recovery			Calculated Head	Measured Head
		Pan Conc. Ag (g/t)	Flotation Conc. Ag (g/t)	Flotation Tailing Ag (g/t)		Gravity % Ag	Flotation % Ag	Overall % Ag	Ag (g/t)	Ag (g/t)
BGF1	79	3182.7	15.00	0.50	14.2	12.5	72.8	85.4	2.93	3.13
BGF2	77	3115.0	14.00	0.50	14.3	13.0	71.7	84.6	2.79	3.13
BGF3	76	3079.2	12.00	0.50	16.7	12.6	72.3	84.9	2.77	3.13
BGF4	78	3415.1	12.00	0.50	13.5	17.0	65.5	82.5	2.47	3.13
Average	**78**	**3198.0**	**13.25**	**0.50**	**14.7**	**13.8**	**70.6**	**84.4**	**2.74**	**3.13**

Source: Inspectorate, 2015

Table 13.3.3.4: Summary of Large Batch Gravity + Flotation Tests Results on the LFZ Master Composite

Balance for Gold										
Test No.	P80 Size (µm)	Assay			Mass (%)	Recovery			Calculated Head	Measured Head
		Pan Conc. Au (g/t)	Flotation			Gravity	Flotation	Overall	Au (g/t)	Au (g/t)
			Conc. Au (g/t)	Tailing Au (g/t)		% Au	% Au	% Au		
BGF5	75	3862.9	7.42	0.07	14.0	32.1	64.2	96.3	1.62	1.55
BGF6	76	3293.0	7.23	0.08	16.9	27.9	68.4	96.3	1.78	1.55
BGF7	79	4692.7	7.25	0.07	17.9	33.3	63.9	97.2	2.03	1.55
BGF8	77	4586.8	6.86	0.07	16.1	35.1	61.6	96.7	1.79	1.55
Average	**77**	**4108.8**	**7.19**	**0.07**	**16.2**	**32.1**	**64.5**	**96.6**	**1.80**	**1.55**
Balance for Silver										
Test No.	P80 Size (µm)	Assay			Mass (%)	Recovery			Calculated Head	Measured Head
		Pan Conc. Ag (g/t)	Flotation			Gravity	Flotation	Overall	Ag (g/t)	Ag (g/t)
			Conc. Ag (g/t)	Tailing Ag (g/t)		% Ag	% Ag	% Ag		
BGF5	75	3867.0	36.00	0.40	14.0	8.8	85.4	94.2	5.89	4.95
BGF6	76	3679.0	30.00	0.50	16.9	9.2	83.9	93.1	6.03	4.95
BGF7	79	3802.0	27.00	0.25	17.9	9.8	86.5	96.3	5.58	4.95
BGF8	77	4083.0	31.00	0.60	16.1	9.2	82.4	91.7	6.03	4.95
Average	**77**	**3857.8**	**31.00**	**0.44**	**16.2**	**9.3**	**84.5**	**93.8**	**5.88**	**4.95**

Source: Inspectorate, 2015

13.3.4 Rougher Flotation Concentrate Upgrading

Rougher flotation concentrates produced during the large-scale flotation tests were used to evaluate collector dosage and regrind requirements during cleaner flotation. Cleaner flotation tests were conducted on "as produced" (without regrinding) rougher concentrates to evaluate reagent dosage requirements. The results of tests conducted on rougher flotation concentrates produced from the UFZ and LFZ master composites are summarized in Table 13.3.4.1 and Table 13.3.4.2 and show that greater than 97% gold recovery and 93% silver recovery was obtained during a single stage of cleaner flotation.

Cleaner flotation tests were conducted on rougher flotation concentrates produced from the UFZ master composite at two regrind sizes of P_{80} 40 and 20 µm. One set of cleaner tests was conducted at natural pH, and a second set of tests was conducted at pH 11, adjusted with hydrated lime, to evaluate the effect of higher pH on cleaner flotation concentrate grade and metal recoveries. The results of these tests are summarized in Table 13.3.4.3 and show that regrinding to P_{80} 20 and 40 µm did not significantly improve cleaner flotation concentrate grade or metal recoveries. Cleaner flotation tests conducted at pH 11 showed no benefit with respect to concentrate grade or gangue mineral depression, but did result in lowering gold recovery to 93% in comparison to the 97% gold recovery achieved at natural pH.

Table 13.3.4.1: Cleaner Flotation Tests versus Reagent Dosage on UFZ Rougher Concentrate

Summary for Gold																
Test No	P80 Size (µm)	Regrind Second	Collector Dosage (g/t feed)		Gold Grade (g/t Au)							Mass (%)	Gold Recovery (%)			
			PAX	A208	Meas. Head	Calc. Head	Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.	Tails		Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.
FC1	61	n/a	8.0	4.0	6.79	7.30	11.4	12.7	18.2	14.5	0.35	49.0	17.7	36.3	84.1	97.6
FC2	61	n/a	4.0	2.0	6.79	6.79	13.3	12.9	14.9	13.7	0.32	48.5	32.8	54.0	95.1	97.6
FC3	61	n/a	2.0	1.0	6.79	7.02	12.5	13.0	15.6	13.9	0.40	49.2	29.8	46.7	94.5	97.1
Summary for Silver																
Test No	P80 Size (µm)	Regrind Second	Collector Dosage (g/t feed)		Silver Grade (g/t Ag)							Mass (%)	Silver Recover (%)			
			PAX	A208	Meas. Head	Calc. Head	Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.	Tails		Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.
FC1	61	n/a	8.0	4.0	16.0	15.2	21.0	22.4	32.9	28.9	2.0	49.0	15.8	30.9	73.3	93.3
FC2	61	n/a	4.0	2.0	16.0	16.6	20.0	31.0	3.8	32.2	2.0	48.5	20.2	53.0	90.3	93.8
FC3	61	n/a	2.0	1.0	16.0	16.7	26.0	30.4	35.1	31.8	2.0	49.2	26.1	45.9	89.5	93.9

Source: Inspectorate, 2015

Table 13.3.4.2: Cleaner Flotation Tests versus Reagent Dosage on LFZ Rougher Concentrate

Summary for Gold																
Test No	P80 Size (µm)	Regrind Second	Collector Dosage (g/t feed)		Gold Grade (g/t Au)							Mass (%)	Gold Recovery (%)			
			PAX	A208	Meas. Head	Calc. Head	Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.	Tails		Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.
FC4	65	n/a	8.0	4.0	7.76	7.84	12.1	11.5	12.3	15.5	0.34	49.3	21.6	32.5	48.0	97.8
FC5	65	n/a	4.0	2.0	7.76	7.72	9.7	10.2	13.7	13.7	0.31	55.5	20.8	41.9	74.2	98.2
FC6	65	n/a	2.0	1.0	7.76	7.93	9.7	10.2	15.0	14.3	0.32	54.5	25.3	39.2	74.2	98.2
Summary for Silver																
Test No	P80 Size (µm)	Regrind Second	Collector Dosage (g/t feed)		Silver Grade (g/t Ag)							Mass (%)	Silver Recover (%)			
			PAX	A208	Meas. Head	Calc. Head	Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.	Tails		Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.
FC4	65	n/a	8.0	4.0	31.0	30.0	50.0	50.4	52.8	57.8	3.0	49.3	23.3	37.1	53.9	94.9
FC5	65	n/a	4.0	2.0	31.0	36.1	46.0	54.1	63.4	52.6	3.0	55.5	21.2	47.6	73.5	96.3
FC6	65	n/a	2.0	1.0	31.0	32.2	35.0	41.6	58.4	56.6	3.0	54.4	22.6	39.3	71.4	95.8

Source: Inspectorate, 2015

Table 13.3.4.3: Summary of Cleaner Flotation Tests Vs Regrind Size

Summary for Gold																	
Test No	P80 Size (µm)	pH	Collector Dosage (g/t feed)		Gold Grade (g/t Au)							Mass (%)	Gold Recovery (%)				
			PAX	A208	Meas. Head	Calc. Head	Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.	Tails		Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.
FC3	61	n/a	2.0	1.0	6.79	7.02	12.5	13.0	15.6	13.9	0.40	49.2	29.8	46.7	94.5	97.1
FC7	40	Natural	2.0	1.0	6.79	6.66	27.6	23.1	19.8	16.4	0.33	39.4	90.0	94.1	95.6	97.0
FC8	22	Natural	2.0	1.0	6.79	6.36	29.7	23.7	17.7	14.6	0.43	41.9	51.9	74.7	85.9	96.1
FC9	40	11.0	2.0	1.0	6.79	6.88	33.2	25.5	21.5	18.1	0.74	35.4	81.7	87.9	90.5	93.1
FC10	22	11.0	2.0	1.0	6.79	6.42	26.3	21.7	17.6	14.2	0.71	42.2	59.7	81.1	87.7	93.6
Summary for Silver																
Test No	P80 Size (µm)	Regrind Second	Collector Dosage (g/t feed)		Silver Grade (g/t Ag)							Mass (%)	Silver Recover (%)			
			PAX	A208	Meas. Head	Calc. Head	Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.	Tails		Cl. Conc. 1	Cl. Conc. 1-2	Cl. Conc. 1-3	Total Cl. Conc.
FC3	61	n/a	2.0	1.0	16.0	16.7	26.0	30.4	35.1	31.8	2.0	49.2	26.1	45.9	89.5	93.9
FC7	40	Natural	2.0	1.0	16.0	12.1	47.0	39.7	34.5	29.1	1.0	39.4	84.5	89.5	92.0	95.0
FC8	22	Natural	2.0	1.0	16.0	13.3	69.0	52.5	38.7	30.3	1.0	41.9	57.7	79.2	89.8	95.6
FC9	40	11.0	2.0	1.0	16.0	14.0	69.0	53.3	44.8	37.7	1.0	35.4	83.6	90.4	93.1	95.4
FC10	22	11.0	2.0	1.0	16.0	12.1	52.0	41.8	33.5	27.4	1.0	42.2	62.3	82.5	88.5	95.2

Source: Inspectorate, 2015

13.3.5 Bulk Cleaner Concentrate Production

Approximately 10 kg of each "as produced" rougher concentrate generated from the large batch gravity + flotation tests was reground to P_{80} of 40 µm, then cleaned once in a Denver 56L flotation cell in order to produce sufficient cleaner flotation concentrates for subsequent cyanidation studies.

The overall results of the large scale tests on 80 kg of material from each master composite (four 20 kg tests on each master composite) are summarized in Table 13.3.5.1. Large-scale gravity + rougher flotation testing resulted in a combined gold recovery of 95.9% from the UFZ master composite and 96.0% gold recovery from the LFZ master composite. This included 31.6% gravity gold recovery from the UFZ composite and 32.6% gravity gold recovery from the LFZ composite. The gravity cleaner concentrate, representing approximately 0.01% of the original mass, contained greater than 4 kg/t Au and 3 kg/t Ag suitable for direct smelting. With a single stage of cleaning, the rougher flotation concentrates, which contributed 15% to 16% of the original mass, was upgraded to cleaner flotation concentrates representing 5.8 to 6.4 wt.% mass and containing 17 to 18 g/t Au and 34 to 72 g/t Ag. After one stage of cleaning, a combined gravity + cleaner flotation gold recovery of about 93% to 94% was achieved.

Table 13.3.5.1: Summary of Large Batch Gravity + Flotation Test Results on the UFZ and LFZ Master Composites

Overall Performance - Gold																	
Composite ID	Primary Grind P80, (µm)	Secondary Grind P80, (µm)	Grade (g/t Au)						Mass Recovery (%)			Gold Recovery (%)			Overall Gold Recovery (%)	
			Calc. Head	Gravity Conc.	1st Cl. Conc.	Ro. Conc.	1st Cl. Tails	Ro. Tails	Gravity Conc.	1st Cl. Conc.	Ro. Conc.	Gravity Conc.	1st Cl. Conc.	Ro. Conc.	Gravity +1st Cl.	Gravity + Ro.
UFZ Master Comp.	78	39.5	1.75	4727	18.5	7.7	0.5	0.09	0.01	5.8	14.7	31.6	61.7	64.2	**93.3**	**95.9**
LFZ Master Comp.	77	40.2	1.78	4109	17.3	7.0	0.3	0.09	0.01	6.4	16.2	32.6	61.9	63.4	**94.5**	**96.0**
Overall Performance - Silver																
Composite ID	Primary Grind P80, (µm)	Secondary Grind P80, (µm)	Grade (g/t Ag)						Mass Recovery (%)			Gold Recovery (%)			Overall Silver Recovery (%)	
			Calc. Head	Gravity Conc.	1st Cl. Conc.	Ro. Conc.	1st Cl. Tails	Ro. Tails	Gravity Conc.	1st Cl. Conc.	Ro. Conc.	Gravity Conc.	1st Cl. Conc.	Ro. Conc.	Gravity +1st Cl.	Gravity + Ro.
UFZ Master Comp.	78	39.5	2.9	3198.0	34.0	14.1	1.0	0.5	0.01	5.8	14.7	13.0	69.1	72.1	**82.1**	**85.1**
LFZ Master Comp.	77	40.2	5.8	3857.8	72.0	30.2	3.0	0.5	0.01	6.4	16.2	9.3	78.5	83.5	**87.8**	**92.8**

Source: Inspectorate, 2015

13.3.6 Concentrate Cyanidation

Concentrate cyanidation tests were carried out on the first-stage cleaner flotation concentrate generated from the two master composites. The leach tests were performed on "as produced" cleaner concentrates at five cyanide concentrations to investigate the impact of cyanide dosages on gold and silver extractions. The optimal cyanide concentration was then used in larger-scale leach tests using the carbon-in-pulp (CIP) procedure, with the resulting leach slurry being subjected to cyanide detoxification studies.

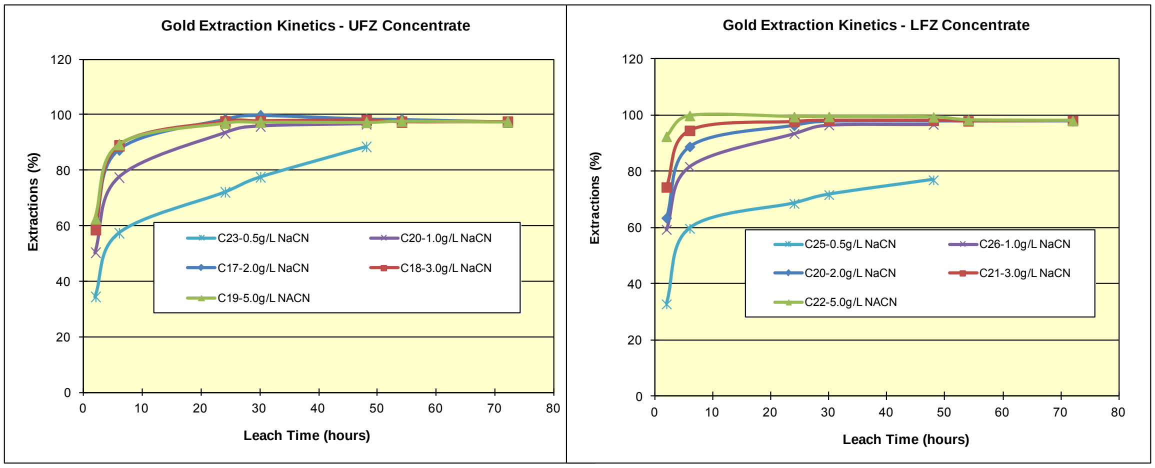
Concentrate cyanidation tests were conducted on 200 g test charges at a slurry density of 30% solids with cyanide concentrations over the range from 0.5 to 5 g/L NaCN. Intermediate solution samples were removed and assayed for leach kinetics. The test results are summarized in Table 13.3.6.1. Gold and silver leach kinetics are plotted in Figures 13.3.6.1 and 13.3.6.2.

Gold extractions of >96% were obtained from cleaner concentrates produced from both master composites at cyanide concentrations ≥1.0 g/L NaCN. Silver was partially refractory with extractions below 50% in all cases. The residual silver appeared to be encapsulated in refractory floatable minerals. Leach kinetics indicate that a 30-hour residence time may be sufficient to leach the concentrate. Although higher cyanide concentrations improved gold and silver extractions, this was at the expense of higher copper dissolution and higher cyanide consumption. After a review of the test results, a dosage of 1.0 g/L NaCN was established to leach the Montagne d'Or concentrate.

Table 13.3.6.1: Summary of Concentrate Cyanidation versus Cyanide Concentration

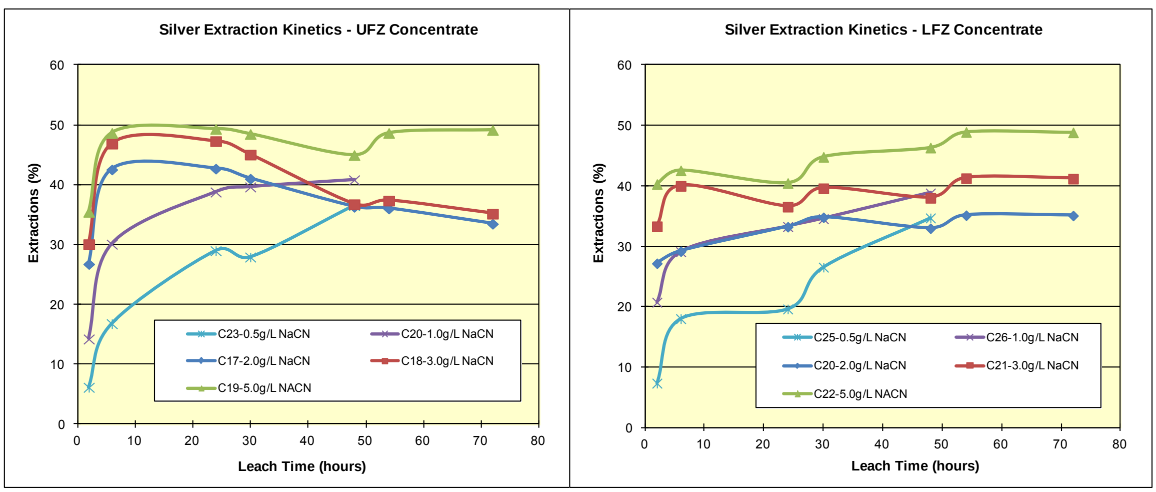
Composite ID	Test No	Test Conditions			Cu in Final PLS (mg/L)	Calculated Head		48-h Extraction		Residue Grade		Consumption (kg/t conc.)	
		% Solids	NaCN (g/L)	Retention		Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Au (g/t)	Ag (g/t)	NaCN	Lime
UFZ Master Composite	C23	30	0.5	48 hours	279	17.40	34.7	88.4	36.6	2.01	22.0	2.79	2.09
	C24	30	1.0	48 hours	357	18.04	35.5	96.6	40.9	0.61	21.0	5.40	1.73
	C17	30	2.0	72 hours	493	18.79	34.6	98.4	36.4	0.50	23.0	8.59	1.50
	C18	30	3.0	72 hours	572	18.08	34.0	98.1	36.8	0.47	22.0	11.39	1.48
	C19	30	5.0	72 hours	679	18.00	35.4	97.3	45.0	0.45	18.0	17.65	1.51
LFZ Master Composite	C25	30	0.5	48 hours	271	19.93	72.0	76.9	34.8	4.60	47.0	3.13	1.97
	C26	30	1.0	48 hours	350	20.24	73.6	96.7	38.9	0.67	45.0	6.01	1.61
	C20	30	2.0	72 hours	532	21.09	72.5	97.9	33.0	0.44	47.0	7.44	0.99
	C21	30	3.0	72 hours	673	20.70	73.2	98.1	38.1	0.40	43.0	12.67	0.98
	C22	30	5.0	72 hours	1033	19.57	74.4	99.3	46.4	0.38	38.0	22.20	1.01

Source: Inspectorate, 2015



Source: Inspectorate, 2015

Figure 13.3.6.1: Gold Extraction versus Leach Retention Time



Source: Inspectorate, 2015

Figure 13.3.6.2: Silver Extraction versus Leach Retention Time

13.3.7 Overall Gravity + Flotation + Cyanidation Performance

The overall response to gravity + flotation + concentrate cyanidation and gold recovery in each process stage is summarized in Table 13.3.7.1. Large-scale gravity + rougher flotation testing with approximately 80 kg of each master composite resulted in a combined gold recovery of 95.9% from the UFZ master composite and 96.0% gold recovery from the LFZ master composite. This included 31.6% gravity gold recovery from the UFZ composite and 32.6% gold recovery from the LFZ composite. The gravity cleaner concentrate, representing approximately 0.01% of the original mass, contained greater than 4 kg/t Au and 3 kg/t Ag suitable for direct smelting. Rougher flotation concentrates containing 15% to 16% of the original mass were upgraded in one stage of cleaner flotation to produce a cleaner flotation concentrate that contained about 17 to 18 g/t Au and

34 to 72 g/t Ag. After one stage of cleaning, a combined gravity + cleaner flotation gold recovery of about 94% Au was achieved.

Cyanidation studies conducted on the cleaner flotation concentrates demonstrated that over 96% of the gold contained in the cleaner flotation concentrate could be extracted after 48 hours of leaching at a cyanide concentration of 1 g/L NaCN. This resulted in an overall gold recovery of 91.2% from the UFZ master composite and 92.5% gold recovery from the LFZ master composite at a cyanide consumption of 5 to 6 kg/t concentrate, equivalent to 0.3 to 0.4 kg/t feed.

Table 13.3.7.1: Overall Gold Recoveries with Gravity + Flotation + Concentrate Cyanidation

Comp. ID	Process Stage	Mass Recovery (%)	Gold Recovery (%)
UFZ Master Comp.	Gravity concentration	0.01	31.6
	Gravity + Rougher flotation	14.67	95.9
	Gravity + Cleaner flotation	5.83	93.3
	Gravity + Cleaner flotation + Concentrate cyanidation (1)	**5.83**	**91.2**
LFZ Master Comp.	Gravity concentration	0.01	32.6
	Gravity + Rougher flotation	16.20	96.0
	Gravity + Cleaner flotation	6.38	94.5
	Gravity + Cleaner flotation + Concentrate cyanidation	**6.38**	**92.5**

(1) at 30 wt.% solids in 1.0 g/L, leach for 48 hours
Source: Inspectorate, 2015

13.4 Metallurgical Testwork: Variability Composites

Gravity + cyanidation and gravity + flotation were evaluated on the ten Montagne d'Or variability composites following the optimal conditions established from the two master composites to evaluate the impact of spatial and grade variations throughout the deposit.

13.4.1 Gravity + Cyanidation

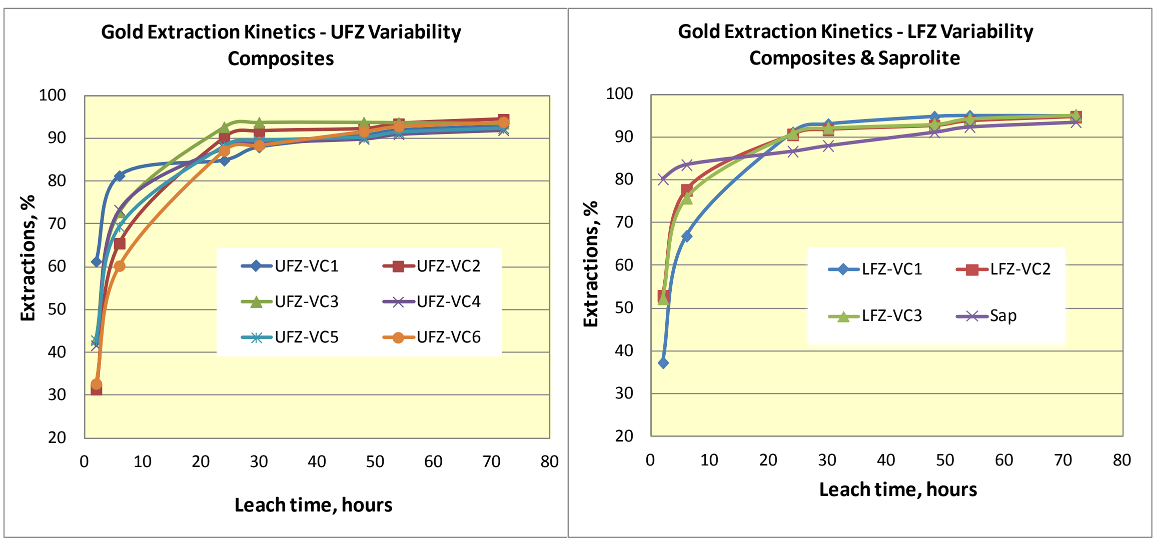
The confirmatory gravity + cyanidation tests were carried out at the optimal grind P_{80} of 75 µm, and a cyanide concentration of 0.5 g/L NaCN during cyanidation of the gravity tailings. The results of these tests are summarized in Table 13.4.1.1 and show that all ten variability composite samples are highly amenable to the gravity + cyanidation process. Gold recovery from the six UFZ variability composites varied from 93.5% to 96.8%, and averaged 95.5% including 35.2% gravity recoverable gold. Gold recovery on the three LFZ variability composites varied from 95.7% to 97.3%, and averaged 96.3% including 28.6% gold in the gravity circuit. Gold recovery of 95.9% was also obtained from the saprolite composite. In addition, over 64% of the silver was recovered with the gravity + cyanidation process, including 20% silver recovery to the gravity cleaner concentrate.

Leach kinetics as shown in Figure 13.4.1.1 suggest leach retention times in the range of 30 to 40 hours. Average cyanide consumption was about 1 kg/t NaCN. Less than 0.5 kg/t lime was required to maintain a pH of 10.5 to 11 during cyanidation.

Table 13.4.1.1: Summary of Gravity + Cyanidation Results on Variability Composites

Test No	Comp. D	Calculated Head		Recovery						Residue Grade		Consumption (kg/t)	
				Gravity		Cyanidation		Overall					
		Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (g/t)	Ag (g/t)	NaCN	Ca(OH)2
GC17	UFZ-VC1	2.57	2.11	39.9	38.5	55.8	37.7	95.7	76.3	0.11	0.50	1.01	0.15
GC18	UFZ-VC2	0.96	1.57	34.0	27.3	62.3	40.8	96.3	68.2	0.04	0.50	1.03	0.81
GC19	UFZ-VC3	3.46	2.41	51.0	40.0	45.8	39.2	96.8	79.3	0.11	0.50	0.93	0.15
GC20	UFZ-VC4	1.08	1.45	19.8	10.8	73.7	54.7	93.5	65.5	0.07	0.50	1.01	0.15
GC21	UFZ-VC5	1.46	2.79	32.6	7.0	62.2	57.2	94.9	64.2	0.08	1.00	1.01	1.05
GC22	UFZ-VC6	3.65	6.96	33.5	21.1	62.4	50.2	95.9	71.3	0.15	2.00	1.11	0.13
UFZ Average		**2.20**	**2.88**	**35.2**	**24.1**	**60.4**	**46.6**	**95.5**	**70.8**	**0.09**	**0.83**	**1.01**	**0.41**
GC23	LFZ-VC1	3.06	9.62	21.0	10.6	74.9	47.8	95.9	58.4	0.13	4.00	1.43	0.15
GC24	LFZ-VC2	0.94	3.88	19.8	3.0	76.0	45.5	95.7	48.5	0.04	2.00	0.94	0.15
GC25	LFZ-VC3	1.31	2.99	45.0	29.5	52.3	37.1	97.3	66.6	0.04	1.00	0.83	0.15
LFZ Average		**1.77**	**5.50**	**28.6**	**14.4**	**67.7**	**43.5**	**96.3**	**57.8**	**0.07**	**2.33**	**1.06**	**0.15**
GC26	Saprolite Variability Comp.	0.97	1.81	36.3	17.2	59.6	27.4	95.9	44.6	0.04	1.00	0.89	1.39
Overall Average		**1.95**	**3.56**	**33.3**	**20.5**	**62.5**	**43.8**	**95.8**	**64.3**	**0.08**	**1.30**	**1.02**	**0.43**

Source: Inspectorate, 2015



Source: Inspectorate, 2015

Figure 13.4.1.1: Gold Extractions versus Leach Retention Time

13.4.2 Gravity + Flotation

Confirmatory gravity + flotation tests were conducted on each of the variability composites at a target primary grind P_{80} of 75 µm and a target secondary grind P_{80} of 40 µm. One stage gravity concentration followed by hand panning was first conducted on ground whole-ore to recover coarse gold. Flotation was then conducted on gravity-scalped tailings to recover the fine gold mainly associated with sulfide minerals. The results of these tests are summarized in Table 13.4.2.1 and show that most of the variability samples responded well to gravity + flotation, with the exception of sample UFZ-VC5 and the saprolite sample, in which a lower slurry pH of 5 to 6 was noted during flotation.

Overall gold recovery from the UFZ and LFZ variability composites into the gravity + cleaner flotation concentrates ranged from 67.5% to 98.7% and averaged 90.9%. Overall gold recovery from the saprolite composite was 69.4%. Overall silver recovery from the UFZ and LFZ variability composites ranged from 54.6% to 93.6% and averaged 71.5%. Silver recovery from the saprolite composite was 29.9%.

Table 13.4.2.1: Summary of Gravity + Flotation Results on Variability Composites

Summary for Gold

Test No	Sample ID	Gold Grade (g/t Au)							Gold Recovery (%)			Mass (%)		
		Meas. Head	Calc. Head	Gravity Cl. Conc.	Flotation Cl. Conc.	Flotation Ro. Conc.	Flotation Cl. Tails	Flotation Tails	Gravity	Gravity+Cl. Flotation	Gravity+Ro. Flotation	Gravity Cl. Conc.	Flotation Cl. Conc.	Flotation Ro. Conc.
GF21	UFZ-VC1	2.07	2.37	543	30.58	15.17	0.28	0.12	22.5	**94.8**	95.5	0.10	5.6	11.4
GF22	UFZ-VC2	0.84	0.83	223	22.64	7.16	0.53	0.10	27.7	**85.6**	88.8	0.10	2.1	7.1
GF23	UFZ-VC3	2.36	3.59	1877	37.10	16.83	0.18	0.04	49.9	**98.7**	99.0	0.10	4.7	10.5
GF24	UFZ-VC4	1.22	1.38	456	19.59	8.59	0.12	0.03	30.1	**97.5**	98.1	0.09	4.8	11.0
GF25	UFZ-VC5	0.93	1.10	249	28.23	7.90	2.12	0.25	23.4	**67.5**	79.1	0.10	1.7	7.8
GF26	UFZ-VC6	2.87	3.60	981	37.24	22.47	0.25	0.09	29.5	**97.5**	97.8	0.11	6.6	10.9
GF27	LFZ-VC1	3.65	3.24	751	22.05	16.26	0.15	0.06	19.9	**98.2**	98.4	0.09	11.5	15.7
GF28	LFZ-VC2	1.29	0.59	167	12.57	4.61	0.17	0.08	24.5	**86.1**	87.6	0.09	2.9	8.1
GF29	LFZ-VC3	1.01	1.08	362	13.69	5.61	0.16	0.08	31.6	**92.4**	93.5	0.09	4.8	11.9
Avg. for Fresh Rock		**1.80**	**1.97**	**623**	**24.85**	**11.62**	**0.44**	**0.09**	**28.8**	**90.9**	**93.1**	**0.10**	**5.0**	**10.5**
GF30	Saprolite Variability Comp.	0.96	0.69	197	9.55	2.30	0.34	0.20	24.6	**69.5**	75.4	0.09	3.2	15.2

Summary for Silver

Test No	Sample ID	Silver Grade (g/t Ag)							Silver Recovery (%)			Mass (%)		
		Meas. Head	Calc. Head	Gravity Cl. Conc.	Flotation Cl. Conc.	Flotation Ro. Conc.	Flotation Cl. Tails	Flotation Tails	Gravity	Gravity+Cl. Flotation	Gravity+Ro. Flotation	Gravity Cl. Conc.	Flotation Cl. Conc.	Flotation Ro. Conc.
GF21	UFZ-VC1	1.7	1.9	73	23	12.3	2.0	0.5	3.7	**70.9**	76.9	0.10	5.6	11.4
GF22	UFZ-VC2	1.6	1.4	50	35	12.6	3.0	0.5	3.6	**56.4**	67.0	0.10	2.1	7.1
GF23	UFZ-VC3	1.7	2.8	691	33	16.0	2.0	0.5	23.7	**79.8**	83.9	0.10	4.7	10.5
GF24	UFZ-VC4	2.3	2.5	50	28	13.9	3.0	1.0	1.9	**56.2**	63.8	0.09	4.8	11.0
GF25	UFZ-VC5	2.0	1.9	50	58	17.5	6.0	0.5	2.8	**56.0**	75.4	0.10	1.7	7.8
GF26	UFZ-VC6	5.8	8.3	1188	99	60.3	2.0	0.5	15.5	**93.6**	94.6	0.11	6.6	10.9
GF27	LFZ-VC1	9.0	9.1	354	68	50.8	3.0	1.0	3.3	**89.4**	90.7	0.09	11.5	15.7
GF28	LFZ-VC2	2.6	2.3	50	41	16.0	2.0	1.0	1.9	**54.6**	59.2	0.09	2.9	8.1
GF29	LFZ-VC3	4.9	5.0	50	90	38.0	3.0	0.5	0.9	**87.0**	91.2	0.09	4.8	11.9
Ave. for Fresh Rock		**3.50**	**3.90**	**284.00**	**52.78**	**26.37**	**2.89**	**0.67**	**6.4**	**71.5**	**78.1**	**0.10**	**5.0**	**10.5**
GF30	Saprolite Variability Comp.	1.3	1.1	50	9	4.3	3.0	0.5	3.8	**29.9**	62.1	0.09	3.2	15.2

Source: Inspectorate, 2015

13.5 Cyanide Detoxification

Two large-scale CIP cyanidation tests were conducted on cleaner flotation concentrates generated from the UFZ and LFZ master composite samples to produce enough feed for continuous cyanide detoxification studies. The large scale leach tests were conducted at a slurry density of 30% solids for 48 hours in baffled tanks with overhead agitation and continuous aeration to maintain >8 mg/L dissolved oxygen at a cyanide concentration of in 1.0 g/L NaCN. After 48 hours of leaching 20 g/L active carbon was added to the leach slurry to produce a barren leach slurry for cyanide detox studies.

The barren cyanidation tailings were subjected to cyanide detoxification using the continuous SO_2/air process. Sodium metabisulfite (MBS) was added as the SO_2 source along with catalytic amounts of $CuSO_4$. Lime was added to control pH at 8.7 throughout the test. An overall target of <1 ppm of total CN in the effluent was established, and the cyanide level was monitored by measuring either free cyanide or weak-acid dissociable cyanide (CNwad) to follow progress at regular intervals. The test results are summarized in Table 13.5.1 show that <0.05 mg/L of total CN and <0.05 mg/L CNwad were achieved on both samples.

As shown in 13.5.2, the usage of SO_2 in the SO_2/Air cyanide detoxification process was 3.1 to 3.8 g SO_2 per gram of total cyanide (4.6 to 5.6 g metabisulfite per gram total cyanide equivalent). Although dissolved copper was present in the concentrate leach solutions, about 0.4 to 0.5 g $CuSO_4$ per gram total cyanide was needed to achieve the reported results.

Table 13.5.1: Summary of Cyanide Detoxification Tests Flotation Concentrate Leach Residues

Items	Unit	UFZ Cl Flotation Conc. Detox Feed	UFZ Cl Flotation Conc. Detox Effluent	LFZ Cl Flotation Conc. Detox Feed	LFZ Cl Flotation Conc. Detox Effluent
Total CN	mg/L	1004	<0.05	974	<0.05
Free CN	mg/L	211	<0.005	206	<0.005
(WAD) CN	mg/L	495	<0.05	455	<0.05
CNO	mg/L	23	777	24	740
SCN	mg/L	735	336	768	393
Cu	mg/L	425	2.13	478	0.96
SO_4	mg/L	1,919	6,702	1,970	7,272

Source: Inspectorate, 2015

Table 13.5.2: Cyanide Detoxification Reagent Consumptions

Test No	Sample ID	TCN Analysis (mg/L)		Reagent Usage (g/g TCN) (1)			
		Before	After	SO_2	$Na_2S_2O_5$ (MBS)	$CuSO_4$	Lime
DT1	UFZ Cl. Conc.	1,004	<0.005	3.1	4.6	0.4	1.8
DT2	LFZ Cl. Conc.	974	<0.005	3.8	5.6	0.5	1.7

(1) Total cyanide
Source: Inspectorate, 2015

13.6 Thickening Studies

Thickening studies were conducted on rougher flotation tailings and on the detoxified cleaner flotation concentrate leach residues generated from the two master composite samples. This work was carried out at Pocock Industrial Inc. in Salt Lake City and documented in their reports dated January 2015 and April 2015.

13.6.1 Flotation Tailings

Testwork conducted on the flotation tailings included:

- Sample characterization to determine the relative size distribution of the material tested;
- Flocculant screening to examine the relative effectiveness of flocculant dose, varying charge, charge density, and molecular weight at known solids concentration, temperature, and pH;
- Static thickening tests to examine flocculation requirement, hydraulic loading rate, unit area requirements, feed solids concentration sensitivity, and predicted underflow solids concentration for the design of conventional thickeners;
- Dynamic thickening tests to examine feed rate (hydraulic stress) versus flocculant dosage, overflow suspended solids, and underflow density at natural pH on each sample; and
- Pulp rheology tests to determine apparent viscosity at known shear rates relative to solids concentration at known temperatures. Based on the observed relationship between shear stress and shear rate, yield values were determined for the pulp in each case.

Various flocculant types possessing an array of molecular weight and surface charge characteristics (from 100% anionic to 100% cationic) were tested during flocculant screening. Hychem AF304, a medium to high molecular weight of 15% charge density, anionic polyacrylamide, produced a slightly more robust floccule structure as compared to the other products tested, and resulted in better overall performance with respect to overflow clarity, decantation rates, and underflow viscosity characteristics.

Under static thickening tests, the tailings samples showed good flocculation and settling characteristics within a feed solids concentration range of 15% to 20%. Both samples required relatively low flocculant dosages (15 to 20 g/t) to achieve reasonable clarity and settling rates. Within the recommended feed solids concentrations and flocculant dosages, unit areas for the flotation and final tailing materials generally fell in the range of 0.15 to 0.21 m^2/Mt/d to achieve the recommended maximum underflow density of 69%.

In dynamic testing, standard in-line flocculation was capable of producing acceptable flocculation efficiency and settling performance in both tailings materials. At optimal conditions, both tailings samples required flocculant dosages slightly higher than those seen in static testing (23 to 26 g/t for both materials tested). The recommended design hydraulic loading rates generally ranged from 4.2 to 4.6 m^3/m^2hr (at the feed densities and flocculant dosages tested). At optimal flocculation conditions, overflow clarities were generally acceptable for both tailing materials (150 to 250 mg/L in testing).

13.6.2 Flotation Concentrate Leach Residues

Testwork conducted on the flotation concentrate leach residues included:

- Sample characterization to determine the relative size distribution of the material tested;
- Flocculant screening to examine the relative effectiveness of flocculant dose, varying charge, charge density, and molecular weight at known solids concentration, temperature, and pH; and

- Static thickening tests to examine flocculation requirement, hydraulic loading rate, unit area requirements, feed solids concentration sensitivity, and predicted underflow solids concentration for the design of conventional thickeners.

Various flocculant types possessing an array of molecular weight and surface charge characteristics (from 100% anionic to 100% cationic) were tested during flocculant screening. Hychem AF304, a medium to high molecular weight of 15% charge density, anionic polyacrylamide, produced a slightly more robust floccule structure as compared to the other products tested, and resulted in better overall performance with respect to overflow clarity, decantation rates, and underflow viscosity characteristics.

Under static thickening tests, the tailings samples showed good flocculation and settling characteristics within a feed solids concentration range of 15% to 20%. Both samples required relatively low flocculant dosages (15 to 20 g/t) to achieve reasonable clarity and settling rates. Within the recommended feed solids concentrations and flocculant dosages, unit areas for the flotation and final tailing materials were generally in the range of 0.23 m^2/Mt/d to achieve the recommended maximum underflow density of about 64% solids.

Dynamic thickening studies were not conducted due to the limited amount of concentrate sample available for testing.

13.7 Relevant Test Results

Table 13.7.1 provides a summary of test results and gold recoveries from relevant tests conducted to evaluate the three process options of:

- Whole-ore cyanidation;
- Gravity concentration followed by cyanidation of the gravity tailings; and
- Gravity concentration followed by flotation and cyanidation of the flotation concentrate.

Although silver recovery was not consistently tracked throughout the metallurgical program, Table 13.7.2 provides a summary of silver recoveries from relevant tests conducted to evaluate:

- Gravity concentration followed by cyanidation of the gravity tailings; and
- Gravity concentration followed by flotation and cyanidation of the flotation concentrate.

It should be noted that reported overall gold and silver recoveries have been reduced by a 2% adjustment factor to allow for gold and silver losses that will occur during commercial operation due to plant inefficiencies.

Table 13.7.1: Estimated Gold Recoveries from Relevant Tests Conducted to Evaluate Three Different Process Options

Process Alternative	**Test**	**Head Grade**		**Gravity**	**Gravity + Ro Flotation**	**Gravity + Cl Flotation**	**Cyanidation**	**Overall**	**Adjustment**	**Adjusted**	**NaCN**	**NaCN**	**Ca(OH)2**
		Calc. Au g/t	**Cu%**	**Au Recovery**	**Au Recovery**	**Unit Au Recovery**	**Au Extraction %**	**Au Recovery %**	**Factor [2]**	**Overall Au Recovery**	**g/L**	**Kg/t**	**Kg/t**
Whole-ore Cyanidation													
UFZ Master	C3	1.42	0.10				95.4	95.4	2	93.4	1.0	1.56	0.20
LFZ Master	C7	2.17	0.13				94.5	94.5	2	92.5	1.0	1.77	0.14
Average		**1.80**	**0.12**				**95.0**	**95.0**	**2**	**93.0**	**1.0**	**1.67**	**0.17**
Gravity + Cyanidation Master Composites													
UFZ	GC9	1.79	0.10	25.9			95.5	96.7	2	94.7	0.5	0.78	0.22
LFZ	GC13	1.80	0.13	27.5			96.2	97.2	2	95.2	0.5	0.92	0.16
Average		**1.80**	**0.12**	**26.7**			**95.9**	**97.0**	**2**	**95.0**	**0.5**	**0.85**	**0.19**
Variability Composites													
UFZ - VC1	GC17	2.57	0.07	39.9			92.9	95.7	2	93.7	0.5	1.01	0.15
UFZ - VC2	GC18	0.96	0.07	34.0			94.5	96.4	2	94.4	0.5	1.03	0.81
UFZ - VC3	GC19	3.46	0.10	51.0			93.5	96.8	2	94.8	0.5	0.93	0.15
UFZ - VC4	GC20	1.08	0.09	19.8			91.9	93.5	2	91.5	0.5	1.01	0.15
UFZ - VC5	GC21	1.46	0.02	32.6			92.4	94.9	2	92.9	0.5	1.01	1.06
UFZ - VC6	GC22	3.65	0.21	33.5			93.8	95.9	2	93.9	0.5	1.11	0.13
LFZ - VC1	GC23	3.65	0.29	21.0			94.8	95.9	2	93.9	0.5	1.43	0.15
LFZ - VC2	GC24	1.29	0.02	19.8			94.7	95.7	2	93.7	0.5	0.94	0.15
LFZ - VC3	GC25	1.01	0.06	45.0			95.2	97.4	2	95.4	0.5	0.83	0.15
Average		**2.13**	**0.10**	**33.0**			**93.7**	**95.8**	**2**	**93.8**	**0.5**	**1.03**	**0.32**
Saprolite	GC26	0.97		36.3			93.5	95.9	2	93.9	0.5	0.89	1.39
Grav + Cl Flot + Cyan Master Composites													
UFZ		1.75	0.10	31.6	95.8	93.2	96.6	91.1	2	89.1	1.0	0.31	0.1
LFZ		1.78	0.13	32.6	96.0	94.5	96.7	92.5	2	90.5	1.0	0.39	0.1
Average		**1.77**	**0.12**	**32.1**	**95.9**	**93.9**	**96.7**	**91.8**	**2**	**89.8**	1.0	0.35	0.1
Variability Composites [1]													
UFZ - VC1	GF21	2.37	0.07	22.5	96.5	94.8	96.5	92.3	2	90.3			
UFZ - VC2	GF22	0.83	0.07	27.7	88.8	85.6	96.5	83.6	2	81.6			
UFZ - VC3	GF23	3.59	0.10	49.9	99.0	98.7	96.5	97.0	2	95.0			
UFZ - VC4	GF24	1.38	0.09	30.1	98.1	97.5	96.5	95.1	2	93.1			
UFZ - VC5	GF25	1.10	0.02	23.4	79.1	67.5	96.5	66.0	2	64.0			
UFZ - VC6	GF26	3.60	0.21	29.5	97.8	97.5	96.5	95.1	2	93.1			
LFZ - VC1	GF27	3.24	0.29	19.9	98.4	98.2	96.5	95.5	2	93.5			
LFZ - VC2	GF28	0.59	0.02	24.5	87.6	86.1	96.5	83.9	2	81.9			
LFZ - VC3	GF29	1.08	0.06	31.6	93.5	92.4	96.5	90.3	2	88.3			
Average		**1.98**		**28.8**	**93.2**	**90.9**	**96.5**	**88.7**	**2**	**86.7**			
Saprolite	GF30	0.69		24.6	75.4	69.5	95.5	67.5	2	65.5			

(1) Concentrate cyanidation extraction for variability composites is based on results obtained from concentrate cyanidation tests from the Master composites
(2) Overall gold recoveries are reduced by 2% to account for process plant inefficiencies
Source: SRK, 2015

Table 13.7.2: Estimated Silver Recoveries

Process Alternative	Test	Calc. Head Ag (g/t)	Gravity Ag Recovery	Gravity + Ro Flotation Ag Recovery	Gravity + Cl Flotation Ag Recovery	Cyanidation Ag Extraction (%)	Overall Ag Recovery (%)	Adjustment Factor (2)	Adjusted Ag Recovery
Gravity + Cyanidation Variability Composites									
UFZ - VC1	GC17	2.11	38.5			61.5	76.3	2	74.3
UFZ - VC2	GC18	1.57	27.3			56.3	68.2	2	66.2
UFZ - VC3	GC19	2.41	40.0			65.5	79.3	2	77.3
UFZ - VC4	GC20	1.45	10.8			61.4	65.6	2	63.6
UFZ - VC5	GC21	2.79	7.0			61.5	64.2	2	62.2
UFZ - VC6	GC22	6.96	21.1			63.6	71.3	2	69.3
LFZ - VC1	GC23	9.62	10.6			53.5	58.4	2	56.4
LFZ - VC2	GC24	3.88	3.0			46.9	48.5	2	46.5
LFZ - VC3	GC25	2.99	29.5			52.6	66.6	2	64.6
Average		**3.75**	**20.9**			**58.1**	**66.5**	**2**	**64.5**
Saprolite	**GC26**	**1.81**	**17.2**			**33.1**	**44.6**	**2**	**42.6**
Grav + Cl Flot + Cyan Master Composites									
UFZ	Bulk	2.90	13.0	85.1	82.1	40.9	41.3	2	39.3
LFZ	Bulk	5.80	9.3	92.8	87.8	38.9	39.8	2	37.8
Average		**4.35**	**11.2**	**89.0**	**85.0**	**39.9**	**40.5**	**2**	**38.5**
Variability Composites (1)									
UFZ - VC1	GF21	1.90	3.7	76.9	70.9	39.9	30.5	2	28.5
UFZ - VC2	GF22	1.40	3.6	67.0	56.4	39.9	24.7	2	22.7
UFZ - VC3	GF23	2.80	23.7	83.9	79.8	39.9	46.1	2	44.1
UFZ - VC4	GF24	2.50	1.9	63.8	56.2	39.9	23.6	2	21.6
UFZ - VC5	GF25	1.90	2.8	75,4	56.0	39.9	24.0	2	22.0
UFZ - VC6	GF26	8.30	15.5	94.6	93.6	39.9	46.7	2	44.7
LFZ - VC1	GF27	9.10	3.3	90.7	89.4	39.9	37.7	2	35.7
LFZ - VC2	GF28	2.30	1.9	59.2	54.6	39.9	22.9	2	20.9
LFZ - VC3	GF29	5.00	0.9	91.2	87.0	39.9	35.3	2	33.3
Average		**3.91**	**6.4**	**78.4**	**71.5**	**39.9**	**32.4**	**2**	**30.4**
Saprolite	**GF30**	**1.10**	**3.8**	**62.1**	**29.9**	**39.9**	**14.2**	**2**	**12.2**

(1) Concentrate cyanidation extraction for variability composites is based on results obtained from concentrate cyanidation tests from the Master composites
(2) Overall gold recoveries are reduced by 2% to account for process plant inefficiencies
Source: SRK, 2015

13.8 Gold Recovery Estimate Assumptions

Table 13.8.1 provides a summary of estimated gold recoveries achievable by each of the process options tested. Gold recovery achievable by a process flowsheet that includes gravity concentration followed by cyanidation is estimated at 95% from the UFZ and LFZ zones and 94% from the saprolite zones.

Gold recovery from a process flowsheet that includes gravity concentration followed by gold flotation from the gravity tailings and cyanide leaching of the flotation concentrate is estimated at 90% for the UFZ and LFZ zones and 65% for the saprolite zones. Estimated gold recoveries have been reduced by a 2% adjustment factor to allow for gold and silver losses that will occur during commercial operation due to plant inefficiencies.

Table 13.8.1: Summary of Estimated Gold Recoveries from Process Options Tested

Process Option	Calc. Head Au (g/t)	Au Extraction (%)	Adjustment Factor	Au Recovery (%)
Whole-ore Cyanidation				
UFZ Master Composite	1.42	95	2	93
LFZ Master Composite	2.17	95	2	93
Gravity + Cyanidation				
UFZ Master Composite	1.79	97	2	95
LFZ Master Composite	1.80	97	2	95
Variability Composite (Average)	2.13	96	2	**94**
Saprolite	0.97	96	2	**94**
Gravity + Flot + Cyan				
UFZ Master Composite	1.75	91	2	89
LFZ Master Composite	1.78	93	2	91
Variability Composite (Average)	1.98	90	2	**88**
Saprolite	0.69	67	2	**65**

Source: SRK, 2015

13.9 Significant Factors

Significant factors include:

- The metallurgical test program was conducted on two master composites formulated from available whole core intervals representing the UFZ and the LFZ, as well as selected variability composites.
- Three process options, including whole-ore cyanidation, a combination of gravity concentration followed by cyanidation of gravity tailing, and gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate, were investigated on two master composites, and the preferred process option and optimal conditions were further verified on ten variability test composites.
- Processing by gravity concentration followed by cyanidation of the gravity tailings yielded the highest overall gold recoveries and was selected at the preferred process option. Gold recovery is projected at about 95% with this process option.

13.10 Process Option Trade-Off Analysis

As part of the study, a trade-off analysis was made between two processing options, namely:

- Option 1: Flotation/Cyanidation of Tailings/Carbon in Leach (CIL); and
- Option 2: Whole-ore CIL.

Option 1 included separate flotation and CIL circuits along with dedicated TSFs with the flotation TSF being unlined and the smaller CIP TSF being lined. Option 2 included only one CIL plant and one large lined TSF. The assumptions for each option are summarized in Table 13.10.1.

Table 13.10.1: Trade-off Analysis Assumptions

	Units	Option 1	Option 2	Variance
Mill Capacity	t/d	12,500	12,500	-
Tonnes Saprolite	kt	4,320	4,320	-
Tonnes Fresh	kt	51,487	51,487	-
Tonnes Processed	Kt	55,807	55,807	-
Saprolite Gold Grade	g/t	2.00	2.00	-
Fresh Gold Grade	g/t	1.79	1.79	-
Total Gold Grade	g/t	1.80	1.80	-
Saprolite Recovery	%	65%	94%	45%
Fresh Recovery	%	90%	95%	5.5%
Overall Recovery	%	87.9%	94.9%	8%
Recovered Gold	koz	2,841	3,069	228
Process Cost	US$/t	$14.95	$16.05	$1.10
Tailings Cost	US$/t	$0.47	$0.47	-
Process Initial Capital (w/ 20% contingency)	US$M	$181.2	$164.4	($16.8)
TSF Initial Capital (w/ 20% contingency)	US$M	$21.2	$33.5	$12.3
TSF Sustaining Capital Cost	US$M	$9.4	$41.9	$32.5
Owners Cost Adj.	US$M	-	-	($0.7)
Total Process Capital (w/ 20% contingency)	US$M	$211.8	$239.9	$27.4

Source SRK, 2015

SRK then undertook an incremental discounted cash flow analysis to determine if Option 2 had better economics than Option 1. The results of the incremental analysis are presented in Table 13.10.2, which shows that Option 2 has better economics at NPV 10% of US$74 million compared to Option 1. On this basis of this analysis, Option 2 was selected as the base case for economic evaluation.

Table 13.10.2: Trade-off Analysis Incremental Discounted Cash Flow Results

Market Prices

Gold	US$/oz	**$1,200**

Physicals

Total Mineralized Material Mined	kt	-
Total Waste Mined	kt	-
Total Material Mined	kt	-
Strip Ratio	w/o	-
Total Mineralized Material Tonnes, Processed	kt	-
Mineralized Material Gold Grade, Processed	g/t	-
Contained Gold, Processed	koz	-
Average Recovery, Gold	%	--
Recovered Gold, Doré	koz	228
Payable Gold, Doré	koz	228

Cash Flow

Total Revenue	**US$000's**	**273,603**
Mining Cost	US$000's	-
Process Cost	US$000's	(61,388)
Tailings Cost	US$000's	-
Site G&A Cost	US$000's	-
Refining/Selling Cost	US$000's	(2,739)
Direct Cash Costs	**US$000's**	**(64,127)**
Royalties	US$000's	(10,673)
Social Responsibility/Community Relations Cost	US$000's	-
Cash Closure & Reclamation Cost	US$000's	-
Other Costs	US$000's	-
Indirect Cash Costs	**US$000's**	**(10,673)**
Total Operating Expense	**US$000's**	**(74,800)**
Operating Margin	**US$000's**	**198,803**
Depreciation Allowance	US$000's	(27,385)
Other Non-Cash Tax Adjustments	US$000's	-
Earnings Before Taxes	**US$000's**	**171,418**
Income Tax	US$000's	(50,283)
Net Income	**US$000's**	**121,135**
Non-Cash Add Back - Depreciation	US$000's	27,385
Other Non-Cash Tax Adjustments	US$000's	-
Working Capital	US$000's	0
Operating Cash Flow	**US$000's**	**148,520**
Initial Capital	US$000's	5,124
Sustaining Capital	US$000's	(32,509)
Closure/Reclamation/Salvage Capital	US$000's	-
Total Capital	**US$000's**	**(27,385)**

Metrics

Economic Metrics (Pre-Tax)		
Free Cash Flow	US$000's	171,418
NPV @ 10%	US$000's	104,665
Economic Metrics (After-Tax)		
Free Cash Flow	US$000's	121,135
NPV @ 10%	US$000's	74,205

Source SRK, 2015

14 Mineral Resource Estimate

14.1 Basis of Resource Estimation

The mineralization at Montagne d'Or is valued primarily for its gold content. There are however, localized zones with significant copper value. Only gold grades were estimated in the work described in this report.

Dr. Bart Stryhas constructed the geologic and Mineral Resource model discussed below. He is responsible for the resource estimation methodology, Mineral Resource classification and resource statement. Dr. Stryhas is independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

The resource estimation is based on the current drillhole database, interpreted lithologies, geologic controls and current topographic data. The resource estimation is supported by drilling and sampling current to April 11, 2015. The estimation of Mineral Resources was completed utilizing computerized resource block model constructed using Vulcan™ modeling software.

14.2 General Geology

The Montagne d'Or deposit is an Archean age gold deposit that has undergone remobilization and shear zone style deformation. The deposit is located within the northern greenstone belt of the Guiana Shield. Mineralization is hosted within the two billion year old, Paramaca Formation composed predominantly of metavolcanic and metasedimentary units. These units have been deformed by folding and ductile shearing which has developed a pervasive foliation striking east-west and dipping steeply to the south. The current model of gold mineralization is a high sulfidization, volcanogenic (VMS) type. Significant portions are thought to have been emplaced as replacement style mineralization. Subsequently, the mineralization has been deformed and partly remobilized within structural controls. Gold mineralization is associated with primary sulfide minerals as replacements within pyrite and chalcopyrite. At a macroscopic scale, the following five types of mineralization have been identified in mapping and drill core logging:

- Semi-massive sulfides (SMS, >20% sulfides) with associated gold mineralization;
- Sulfides as disseminations and stringers with associated gold mineralization;
- Late-stage disseminated euhedral pyrite mineralization;
- Rhythmic mafic tuff with associated pyrrhotite mineralization; and
- Gold mineralization associated with quartz veins.

14.3 Controls on Gold Mineralization

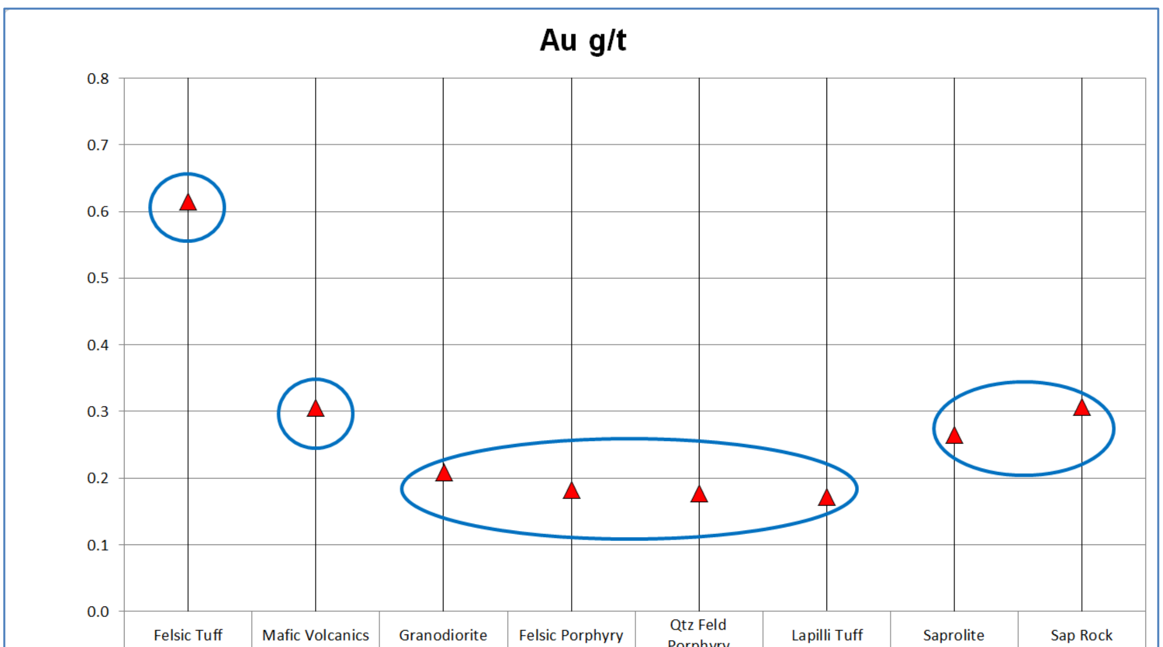
Gold mineralization is controlled mainly by structural fabric and lithology. The mineralization is localized in planar zones which have recurrent distribution and highly variable grades. Anomalous gold grades typically occur in zones 3 m to 10 m wide which are separated by barren or lower grade zones 10 m to 30 m wide. This is a common occurrence in these types of deposits and it is very important to consider this fact when designing reliable resource estimation. The orientation of this preferred plane of mineralization has been identified and refined over the past several years of drilling. Columbus has recently undertaken a program of oriented core drilling which has provided valuable information to better understand the structural geology of the deposit. All structural orientation data to date was acquired and plotted on lower hemisphere stereonets. The structural fabric data includes; foliation, shear planes, lithologic contacts and veins. The results of the stereonet plots are summarized in Table 14.3.1 and the actual stereonets are presented in Appendix A. The preliminary results confirm that the preferred orientation of mineralization as interpreted by Columbus, does follow along the average foliation and shear planes.

Table 14.3.1: Average Orientations of Structural Fabrics

Fabric	Strike	Dip °	# Measurements
Foliation	N86E	-70S	1,119
Shear Planes	N90E	-74S	35
Contacts	N83E	-70S	785
Veins	N87E	-71S	878

Source: SRK, 2015

As part of the most recent drilling campaign, most of the historic core was re-logged to create a unified system of lithologic descriptions. This has resulted in a detailed, 3-D geologic model created by using ARANZ Leapfrog® Geo software (Leapfrog®). To illustrate the importance of lithologic control of mineralization, SRK constructed a box plot of gold values in the drillholes database subdivided by lithology. The results are presented in Figure 14.3.1. The box plot shows four relative levels of mineralization controlled by lithology. Each of these four lithic types or groups were geologically modelled and estimated independently.



	Felsic Tuff	Mafic Volcanics	Granodiorite	Felsic Porphyry	Qtz Feld Porphyry	Lapilli Tuff	Saprolite	Sap Rock
Q1	0.030	0.010	0.010	0.010	0.012	0.030	0.020	0.018
Min	0.000	0.000	0.000	0.001	0.000	0.002	0.000	0.000
Median	0.090	0.020	0.028	0.020	0.032	0.051	0.055	0.050
▲Mean	0.616	0.306	0.208	0.182	0.178	0.173	0.265	0.307
−Max	163.000	70.990	62.500	31.260	9.560	11.074	29.154	81.400
Q3	0.318	0.084	0.080	0.060	0.100	0.100	0.177	0.160
NSamples	21188	8918	11678	997	977	858	5598	2004

Source: SRK, 2015

Figure 14.3.1: Box Plot of Gold Grade by Lithology

14.4 Density

Density testing was performed on the drill core during 2007 and from 2011 to 2014, a total of 3,323 density measurements were taken from all lithic varieties by onsite personnel. The averages of each lithology are listed in Table 14.4.1. These densities were assigned in the block model based on the lithology of the block.

Table 14.4.1: Densities Assigned in the Block Model

Rock Type	Number of Measurements	Average Density g/cm^3
Saprolite	354	1.695
Saprolite-Rock Transition	193	2.365
Felsic Tuff	1,056	2.911
Mafic Volcanics	413	3.154
Granodiorite	615	2.754
Feldspar Porphyry	61	2.786
Quartz-Feldspar Porphyry	164	2.817
Lapilli Tuff	75	2.864
Diabase Dikes	392	3.016

Source: SRK, 2015

14.5 Sample Database

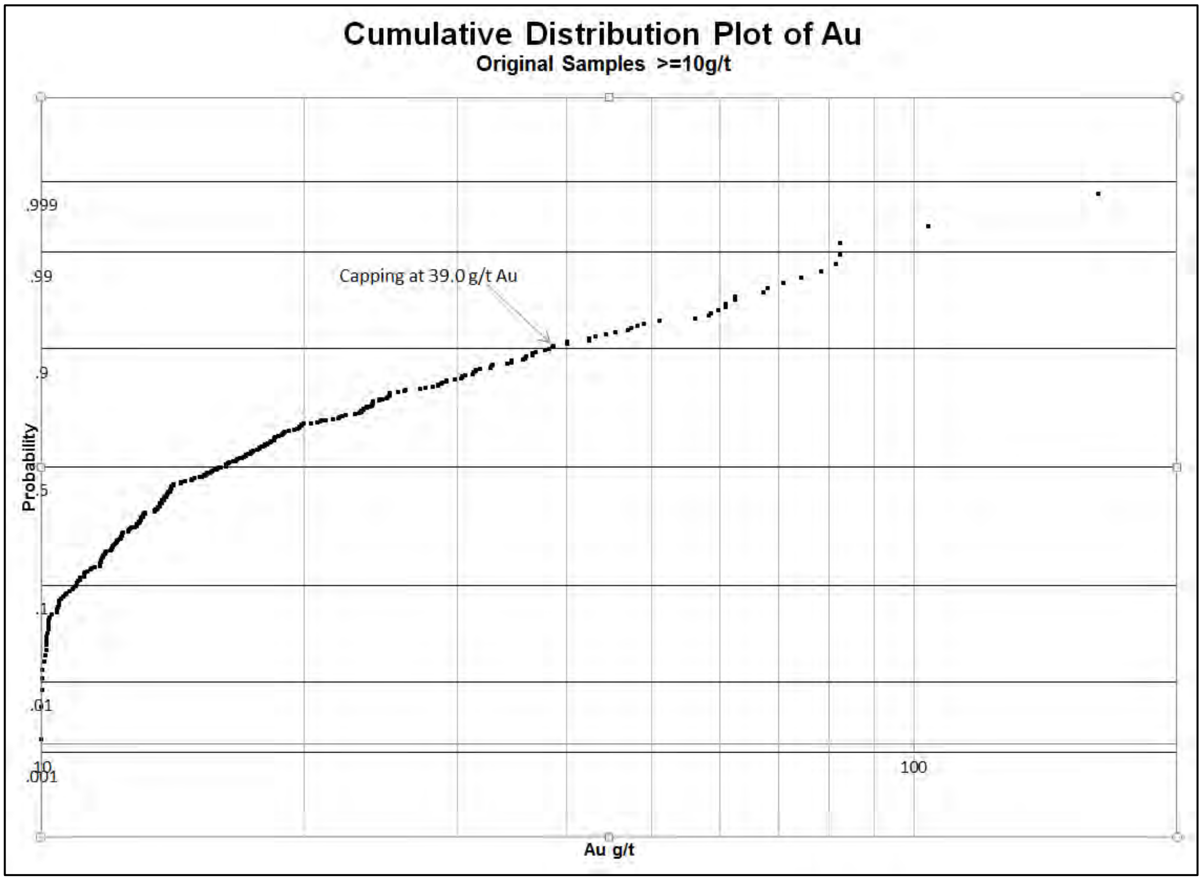
The April 11, 2015 database contains information from 224 diamond drillholes and 37 channel samples. The drilling was completed in two main campaigns. A previous owner drilled 56 holes between 1996 and 1998. Columbus completed an additional 168 holes from 2011 to April, 2015. The channel samples were all collected from surface between 1995 and 1997. SRK has previously reviewed the 1995 through 1998 exploration data and found it to be of sufficient quality to support an industry standard, resource estimation.

The database includes four Excel® files containing information on collar locations, downhole surveys, lithology and gold assays. There are 49,513 valid entries in the assay file with an average sample length of 1.03 m.

14.6 Capping and Compositing

The original drillhole gold values were assessed for statistical outliers using a lognormal cumulative distribution plot and decile analysis. The decile analysis was used to identify the appropriate bin range for capping and the cumulative distribution plot was used to define the final capping level. The results of the cumulative distribution plot are presented in Figure 14.6.1. The Au capping level was chosen at 39 g/t mainly because this is the point where the cumulative distribution trends lose continuity and the data values above, show irregular distribution. The Au capping resulted in 25 samples ranging from 40.1 g/t to 163 g/t being reduced to 39 g/t prior to compositing. This was a net loss of 3% of all gold in the database.

Compositing was completed in 3 m downhole lengths with no breaks at lithologic contacts. The 3 m length was chosen as an appropriate size for two reasons. This length includes three original assay intervals so that it provides some smoothing of the data while still preserving the recurrent nature of the gold mineralization. The 3 m composite length also results in approximately two composites being included within the diagonal intersection of the 5 m, Y direction block size.



Source: SRK, 2015

Figure 14.6.1: Log Normal Cumulative Distribution Plot of Gold Assays above 10 g/t

14.7 Block Model

The block model limits of the SRK resource estimations are listed below. The block dimensions are based on a compromise between the average drillhole spacing, a typical open pit selective mining unit, the variability of the mineralization and computational efficiency of keeping the model under ten million blocks. The block model limits and block sizes are listed in Table 14.7.1. There are 7,086,240 blocks in the model.

Table 14.7.1: Block Model Size and Extents

Orientation	Minimum (m)	Maximum (m)	Block Dimension (m)
Easting	172,200	175,160	10
Northing	520,200	521,150	5
Elevation (AMSL)	-150	480	5

Source: SRK, 2015

14.8 Estimation Strategy

Columbus constructed Leapfrog® software generated wireframe solids which enclose anomalous gold mineralization at a 0.3 g/t Au threshold. The grade shell was checked for validity using two methods. First, it was queried to determine how many samples within it fall above the 0.3 g/t threshold. The query showed that 79% of the samples within the grade shell were above the threshold. Next, it was visually inspected to be sure the geometry was reasonable, based on the nearby drillholes. Four rock types/groups were used as shown in Section 14.3, Figure 14.3.1. Each rock type/group was estimated independently both internal and external to the grade shell using only samples from the same domain. The resultant grade estimation was therefore conducted in eight domains. As discussed in Section 14.3, the gold mineralization is strongly controlled by thin planar zones. These generally strike east-west and dip approximately -68° south. To estimate metal grades along this orientation, trend planes were constructed which mark the hangingwall and footwall to the significant mineralization. The search ellipsoid used for each model block paralleled these trend surfaces. This creates a dynamic search anisotropy which varied according to the average orientation of the shear zone throughout the block model. An Inverse Distance Weighting Squared (IDW^2) algorithm was used for the grade estimations since the variograms have very high nugget values and short ranges.

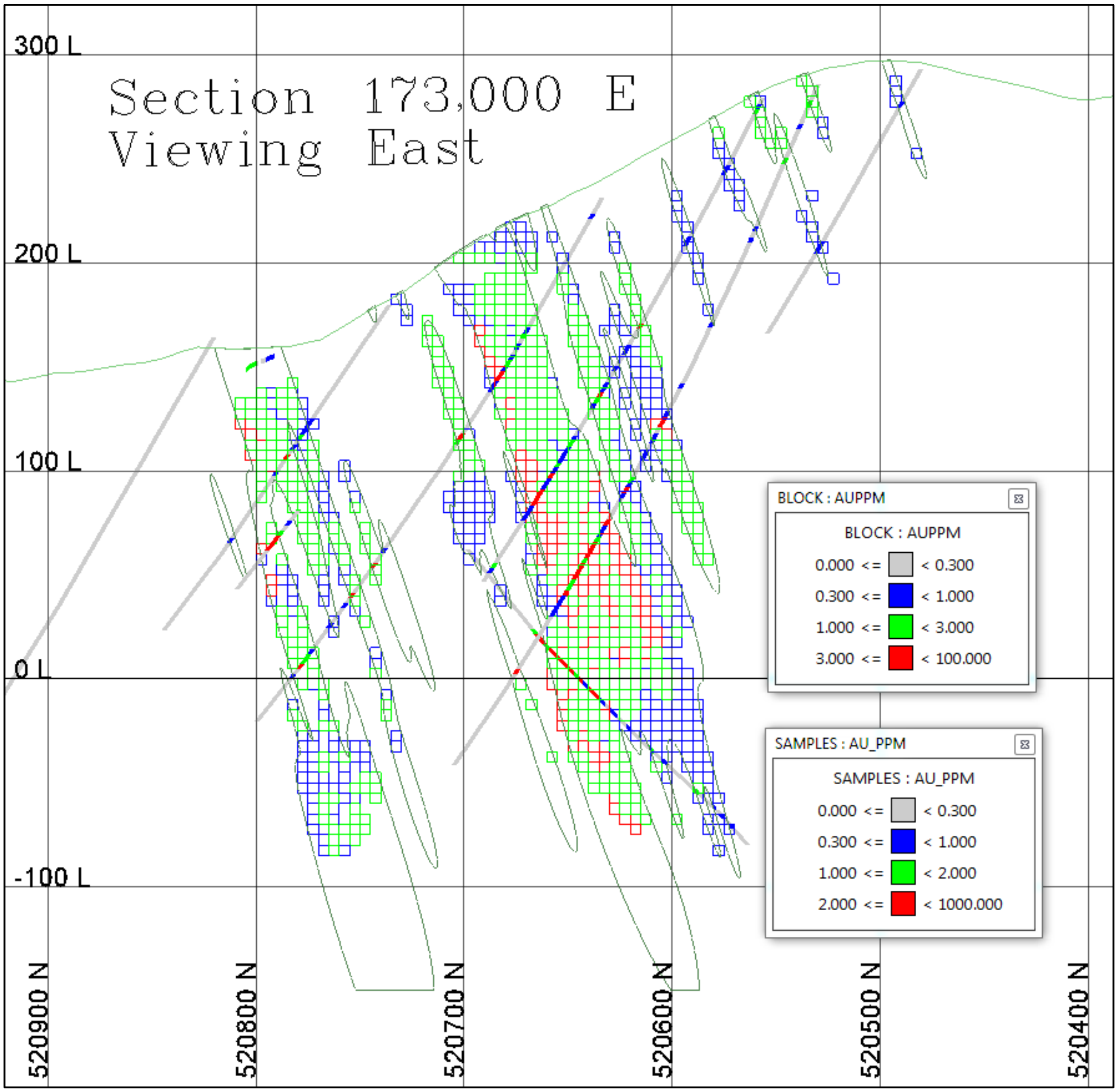
14.9 Estimations Procedures

The grade estimations for all metals in all domains utilize a four pass sample search strategy with each pass searching longer distances than the previous. In each domain, all blocks located within 75 m to the closest sample were identified and grade was only estimated in these blocks. Because the grade shell and distance restriction has been predetermined; and mineralized blocks are now isolated from less-mineralized blocks, the model is allowed to search relatively long distances in the preferred plane of mineralization and the direction normal to it. This method provides for a larger pool of composites to be considered resulting in appropriate grade smoothing. The search distances and sample selection criteria are listed in Table 14.9.1. Sample length weighting is used in all estimations to account for any short composites located at the ends of drillholes. As part of the grade estimation, model validation is conducted as an interactive process. To achieve proper validation, some higher grade composites were limited by the distance they could be interpolated. A high-grade composite restriction, as listed in Table 14.9.1, means that any sample above the listed grade could only be interpolated over the listed distance. Figures 14.9.1 and 14.9.2 show representative cross sections of the gold and copper estimation results.

Table 14.9.1: Au Grade Estimation Parameters

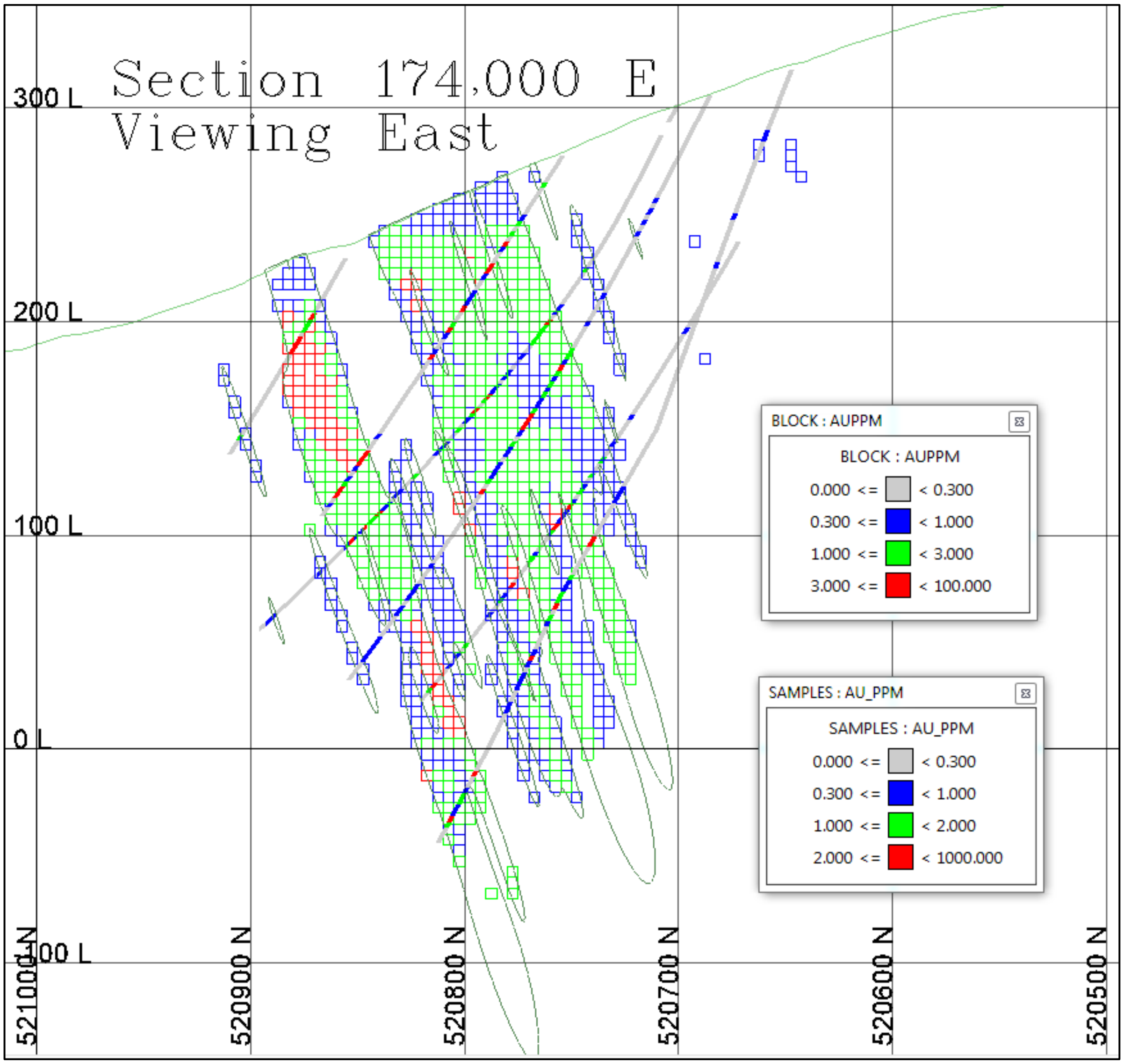
Estimation	Estimation Pass	Search Range (x,y,z) m	Min/Max Samples	Octant Restriction	High Grade Composite Restriction(grade. x, y, z distances)
Saprolite/Sap Rock Inside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	35,35,5	3/8	2 Samp/Octant	None
	3	65,65,10	3/8	2 Samp/Octant	>5.5 g/t <35 m, 35 m, 5 m
	4	125,125,15	3/8	2 Samp/Octant	>5.5 g/t <35 m, 35 m, 5 m
Saprolite/Sap Rock Outside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	50,50,25	1/8	2 Samp/Octant	
Felsic Tuff Inside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	35,35,5	3/8	2 Samp/Octant	None
	3	65,65,10	3/8	2 Samp/Octant	>15 g/t <35 m, 35 m, 5 m
	4	125,125,15	3/8	2 Samp/Octant	>15 g/t <35 m, 35 m, 5 m
Felsic Tuff Outside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	25,25,5	1/8	2 Samp/Octant	
Mafic Volcanics Inside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	35,35,5	3/8	2 Samp/Octant	
	3	65,65,10	3/8	2 Samp/Octant	>9.0 g/t <35 m, 35 m, 5 m
	4	125,125,15	3/8	2 Samp/Octant	>9.0 g/t <35 m, 35 m, 5 m
Mafic Volcanics Outside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	25,25,5	1/8	2 Samp/Octant	
Other Lithologies Inside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	35,35,5	3/8	2 Samp/Octant	None
	3	65,65,10	3/8	2 Samp/Octant	>6.0 g/t <35 m, 35 m, 5 m
	4	125,125,15	3/8	2 Samp/Octant	>6.0 g/t <35 m, 35 m, 5 m
Other Lithologies Outside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	25,25,5	1/8	2 Samp/Octant	

Source: SRK, 2015



Source: SRK, 2015

Figure 14.9.1: Representative Cross Section 173,000E with Estimated Au Grades



Source: SRK, 2015

Figure 14.9.2: Representative Cross Section 174,000E with Estimated Au Grades

14.10 Model Validation

Five techniques were used to evaluate the validity of the block model. First, the interpolated block grades were visually checked on sections, plan views and in 3-D for comparison to the composite assay grades. Second, the general model estimation parameters were reviewed to evaluate the performance of the model with respect to supporting data. This included the number of composites used, number of drillholes used, average distance to samples used, and the number of block estimated in each pass. The results of this analysis are presented in Table 14.10.1. Third, statistical analyses were made comparing the estimated block grades from the IDW^2 estimation to the composite sample data supporting the estimation. Table 14.10.2 lists the results of the statistical comparison. In all cases, the block grades are very close to, or slightly below, the composite grades as desired. Fourth, a nearest neighbor estimation was run using a single composite to estimate each block using the same parameters as the IDW^2 estimation. The total contained metal, at a zero CoG in the nearest neighbor estimation, is compared to the IDW^2 estimation at the same cut-off. The results of this comparison are listed in Table 14.10.3. The final validation was to construct N-S oriented swath plots located every 50 m spacing. The results shown in Figure 14.10.1 illustrate strong correlation between block grades and composites with an appropriate amount of smoothing.

Table 14.10.1: Estimation Performance Parameters of Au Estimation in Grade Shell

Estimation	Samples Used (#)	Drillholes Used (#)	Average Distance to Samples (m)	Blocks Estimated (%)
Pass 1	1.4	1	2.7	2
Pass 2	4.1	2.3	21	53
Pass 3	4.7	2.6	38	22
Pass 4	5.3	3.0	69	23
All Passes	4.5	2.5	35	100

Source: SRK, 2015

Table 14.10.2: Model Validation Statistical Results in Grade Shell

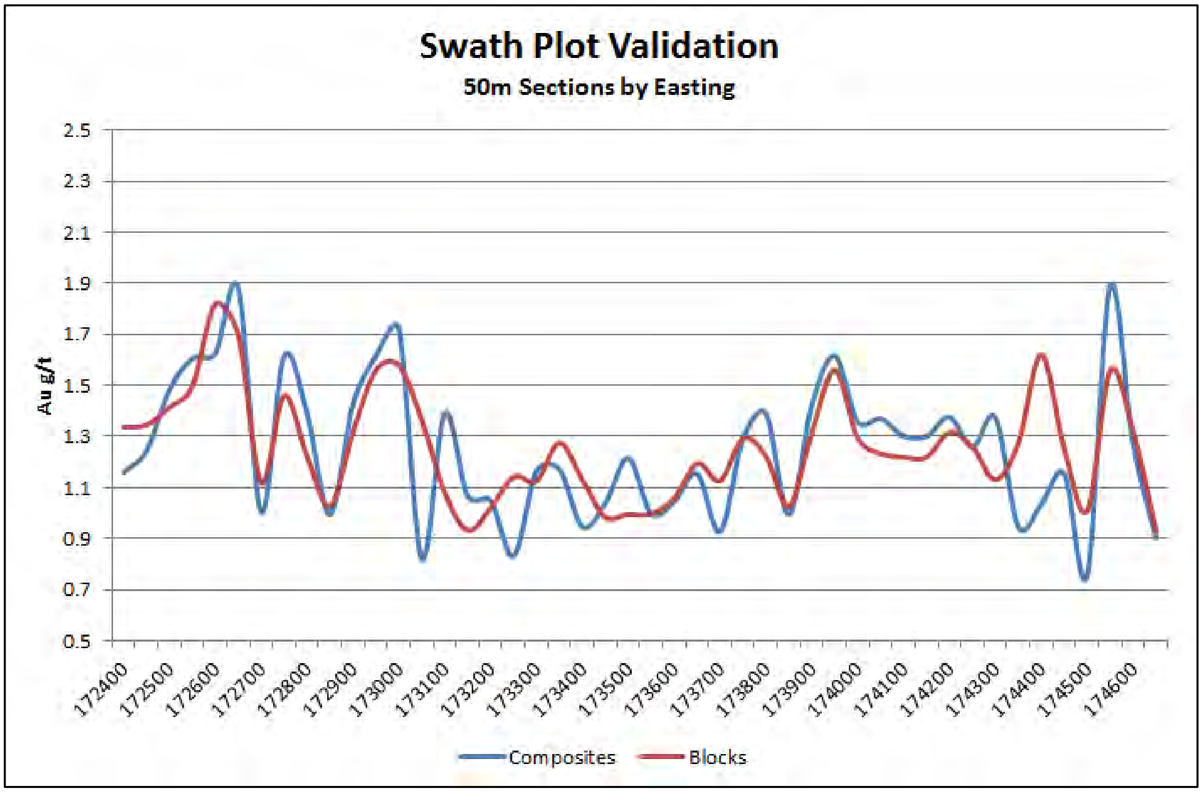
Estimation	Average Composite Grade (g/t)	Average Block Grade (g/t)	Difference of Composites to Blocks (%)
Saprolite/Sap Rock	0.932	0.851	8.7
Felsic Tuff	1.479	1.389	6.1
Mafic Volcanics	1.306	1.270	2.8
Other Lithologies	0.804	0.780	3.0
All Lithologies	1.263	1.255	0.6

Source: SRK, 2015

Table 14.10.3: Model Validation nearest Neighbor Results in Grade Shell

Estimation	Cut-off (g/t)	Tonnes (M)	IDW^2 Grade (g/t)	NN Au Grade (g/t)	% Difference of Metal Mass, IDW^2 to NN
Saprolite/Sap Rock	0	8.3	0.8527	0.8194	3.9
Felsic Tuff	0	87.4	1.3891	1.3936	-0.3
Mafic Volcanics	0	19.7	1.2698	1.2748	-0.4
Other Lithologies	0	13.4	0.7799	0.7544	3.3
All Lithologies	0	128.8	1.2729	1.2719	0.1

Source: SRK, 2015



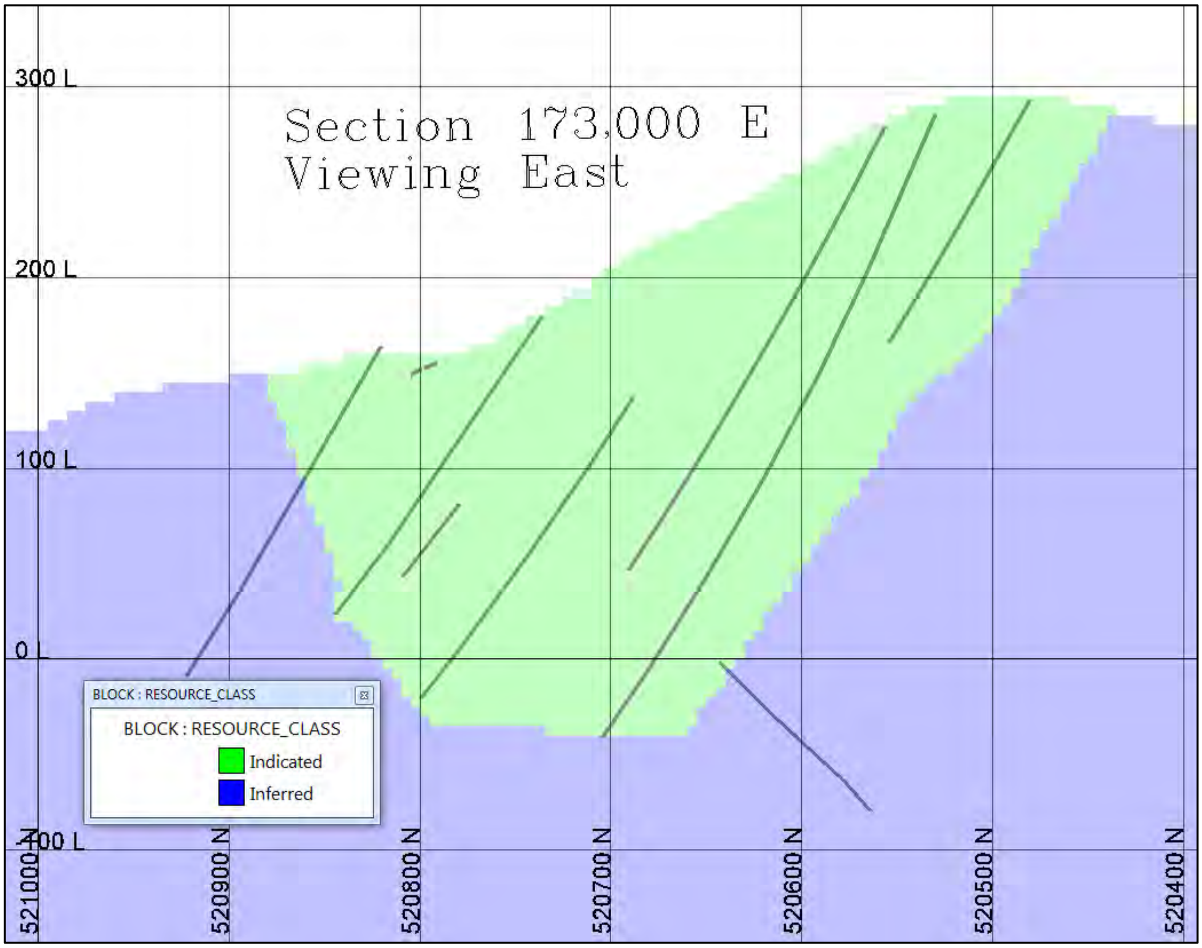
Source: SRK, 2015

Figure 14.10.1 North-South Oriented Swath Plots

14.11 Resource Classification

Mineral Resources are classified under the categories of Measured, Indicated and Inferred according to CIM guidelines. Classification of the Mineral Resources reflects the relative confidence of the grade estimates and the continuity of the mineralization. This classification is based on several factors including sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization, data verification to original sources, specific gravity determinations, accuracy of drill collar locations, accuracy of topographic data, quality of the assay data and many other factors which influence the confidence of the mineral estimation. No single factor controls the Mineral Resource classification, rather each factor influences the end result.

The Mineral Resources reported for the Montagne d'Or deposit are classified as Indicated and Inferred Mineral Resources. This is based primarily on drillhole spacing since all other supporting data is of good quality. Wire frame solids were constructed around the areas where the average drillhole spacing is approximately 50 m or less and these were used to assign the Indicated Mineral Resource classification. All blocks outside of these wireframes were classified as Inferred Mineral Resources. Figure 14.11.1 shows a representative cross section of the Mineral Resource classification.



Source: SRK, 2015

Figure 14.11.1: Representative Cross Section of Resource Classification

14.12 Mineral Resource Statement

The Montagne d'Or Mineral Resource statement is presented in Table 14.12.1. The resource is confined within a Whittle ™ optimization pit shell and a CoG of 0.4 g/t Au applied. The pit shell and CoG assumes open-pit mining methods and is based on a mining cost of US$1.50/t, milling cost of US$15/t, administration cost of US$1/t, a gold price of US$1,300/oz., 90% gold recovery, gold refining cost of US$8/oz, and 5% NSR royalty. A 45° pit shell slope was used for bedrock and a 35° pit shell slope was used for saprolite. The reported Mineral Resources include material from all estimation domains.

Table 14.12.1: Montagne d'Or Mineral Resource Statement as of April 11, 2015 SRK Consulting (U.S.), Inc.

Classification	Au Cut-Off (g/t)	Tonnes (M)	Au (g/t)	Contained Au (M oz)
Indicated	0.40	83.24	1.455	3.893
Inferred	0.40	22.37	1.550	1.115

- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- All figures rounded to reflect the relative accuracy of the estimates.
- Metal assays were capped where appropriate.
- Mineral Resources are reported based on a CoG of 0.4 g/t Au, and are reported inside a conceptual pit shell based on appropriate mining and processing costs and metal recoveries for oxide and sulfide material.
- CoGs are based on a mining cost of US$1.50/t, milling cost of US$15/t, administration cost of US$1/t, a gold price of US$1,300/oz., 90% gold recovery, gold refining cost of US$8.00/oz, and 5% NSR royalty.

Source: SRK, 2015

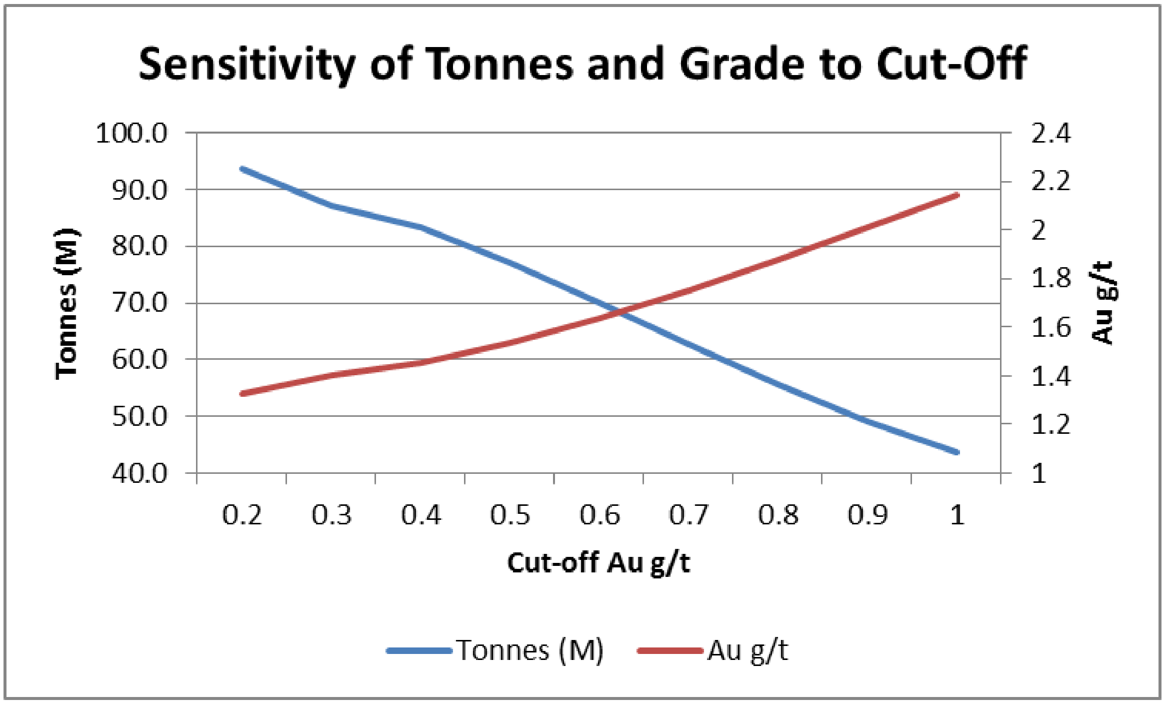
14.13 Mineral Resource Sensitivity

The Mineral Resources shown in Table 14.13.1 are presented at a range of CoGs, subdivided by resource classification. Graphical representations of the grade and tonnage sensitivities of the Indicated resources are presented in Figure 14.13.1. All resources are confined within the Whittle™ optimization pit shell.

Table 14.13.1 Mineral Resource Sensitivity [(1)]

Indicated			
Cut-off	**Tonnes (M)**	**Au (g/t)**	**Au (M oz)**
0.2	93.7	1.325	3.99
0.3	87.2	1.405	3.94
0.4 [(2)]	83.2	1.455	3.89
0.5	77.1	1.536	3.81
0.6	70.1	1.634	3.68
0.7	62.5	1.753	3.53
0.9	55.6	1.878	3.36
1.0	49.1	2.013	3.18
Inferred			
Cut-off	**Tonnes (M)**	**Au (g/t)**	**Au (M oz)**
0.2	24.2	1.455	1.13
0.3	23.1	1.510	1.12
0.4 [(2)]	22.4	1.550	1.11
0.5	21.3	1.605	1.10
0.6	19.8	1.683	1.07
0.7	18.2	1.773	1.04
0.9	16.5	1.883	1.00
1.0	14.8	1.998	0.95

(1) Tonnes and grade have been rounded to reflect the level of expected accuracy.
(2) Base Case CoG.
Source: SRK, 2015



Source: SRK, 2015

Figure 14.13.1: Sensitivity of Tonnes and Grade to Cut-off

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

15 Mineral Reserve Estimate

There are currently no Mineral Reserves for the Project, based on the current level of study.

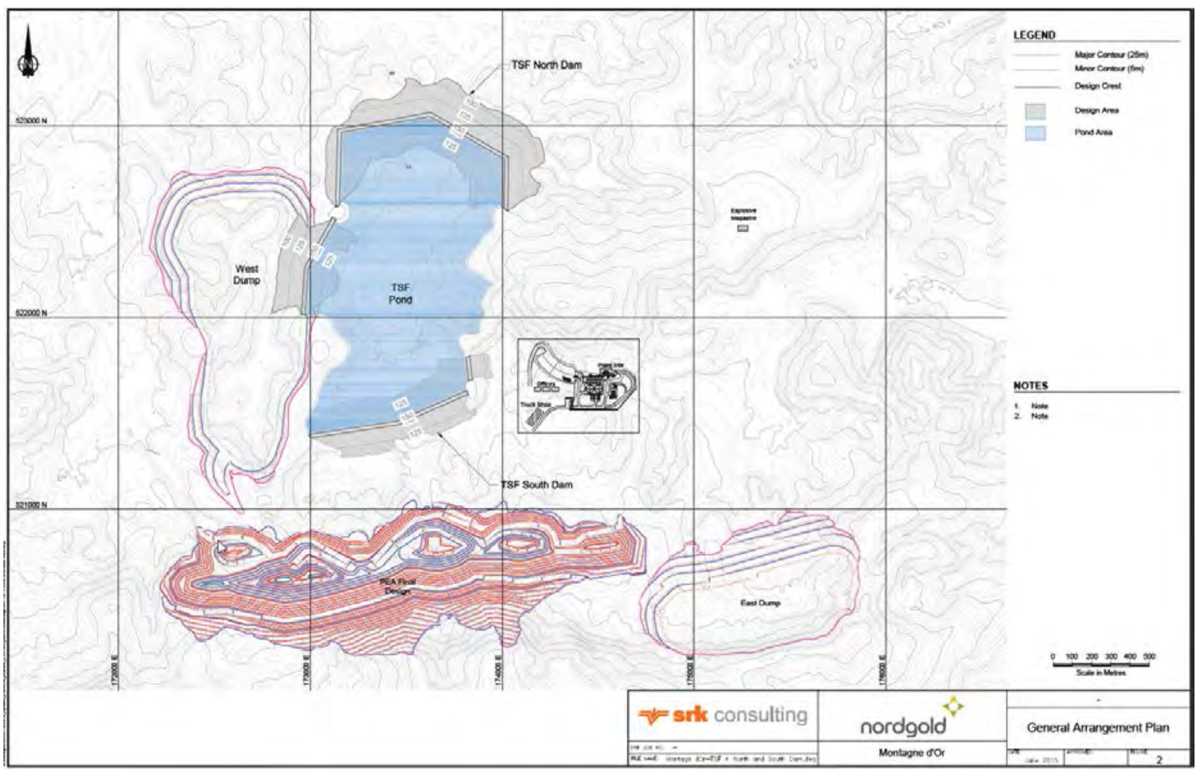
16 Mining Methods

16.1 Introduction

Montagne d'Or in French Guiana is located on the side of a moderately sized hill, surrounded by dense tropical rainforest in a remote location that has been disturbed by garimpeiro miners. Recent exploration programs have successfully confirmed mineralization along strike of the deposit resulting in approximately 5 Moz potentially available for extraction.

The Montagne d'Or mine will be an open pit mine that uses gravity/cyanidation as the primary method of extracting gold from the Mineral Resource. Through the process of pit optimization, pit design, production scheduling, and capital and operating cost estimation, the resource used in the PEA resulted in an estimated 3.23 Moz Au defined in situ before metallurgical recoveries.

The PEA open pit is approximately 2.5 km long by 500 m wide and 400 m deep with a total volume of 127.7 Mm^3 with a stripping ratio of 5 t of waste for every tonne of mill feed. Figure 16.1.1 illustrates the pit design, dump design and expected tailings location for the Project.



Source: SRK 2015

Figure 16.1.1 Montagne d'Or Site Layout

The mine production schedule is based on feeding the processing facility operating at a rate of 12,500 t/d or approximately 4.5 Mt/y of mill feed. The mill feed is broken into three CoGs that represent the internal CoG if gold was US$400/oz, US$800/oz and US$1,200/oz, and that includes a 90% recovery for all rock types for the purpose of the CoG calculations.

The planned mining rate targets approximately 80,000 t/d, which provides more mill feed than is needed to be put into the plant, and therefore mill feed stockpiles will be used to store the excess. The use of stockpiles ensures that the highest grade mill feed is sent to the crusher before lower grade is processed. This creates a variable cut-off that defers marginal mill feed that will be processed at the end of the mine life, thus optimizing the Project NPV and cash flow. The maximum stockpile size is approximately 8 Mt of material.

Although the Montagne d'Or project is a PEA and the inclusion of Inferred material is permitted, it should be noted that there is only 6% Inferred material being sent to the mill or alternately 3.2 Mt mill feed above a 0.7 g/t Au CoG.

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

16.2 Mine Block Model

The resource block model has been modified to store information necessary for the mine production schedule, fleet estimation, Whittle™ pit optimization and resource calculation. Table 16.2.1 details the base variables used in the block model and translation fields if the variables are text.

Table 16.2.1: Mine Block Model Variables

Variables	Default	Type	Description
au_ppm_gs	0	double	Resource Grade
lith	0	integer	Resource Lithology – "brock"
density	0	double	Resource Density
rescat	0	integer	Resource Classification
bau	0	float	MI Au Grade
baui	0	float	MII Au Grade
bden	0	float	Whittle™ Density
r_den	0	float	Resource Density
btopo	100	float	Topo Variable
berm	0	float	Berm Width
batter	0	float	Batter Angle (Bench face angle)
ore_cyc	0	float	Ore Cycle Time
ore_dist	0	float	Ore Cycle Distance
waste_cyc	0	float	Waste Cycle Time
waste_dist	0	float	Waste Cycle Distance
lg_cyc	0	float	Low Grade Cycle Time
lg_dist	0	float	Low Grade Cycle Distance
total_cyc	0	float	Total Cycle Time
total_dist	0	float	Total Distance
haul_cut	waste	Text	waste = 0 ore = 1
brock	air	Text	air = 0 mets = 1 dike = 2 amps = 3 fltf = 4 flpy = 5 lptf = 6 gran = 7 mfvl = 8 qtzp = 9 sapr = 10 sap = 11
product	waste	Text	waste = 0 mg800 = 1 hg400 = 2 lg1200 = 3

Source: SRK, 2015

The calculation of grade bins, slope angles, resource cut-offs and other manipulations to the resource block model require that a series of block scripts are run. Table 16.2.2 details the scripts and logic for the creating of mine block variables used in the mine planning process.

Table 16.2.2: Block Model Scripts

Task	Script	Description
Whittle™	Density for topography	bden = r_den * btopo / 100

Whittle™	if (lith eq 1) then
	brock = "sap"
	endif
		Create measured Indicated and Inferred gold variable

Whittle™	if (lith eq 2) then
	brock = "sapr"
	endif
		Name variable for Saprolite

Whittle™	if (lith eq 3) then
	brock = "fltf"
	endif
		Name variable for Felsic Tuff

Whittle™	if (lith eq 4) then
	brock = "flpy"
	endif
		Name variable for Feldspar Porphyry

Whittle™	if (lith eq 5) then
	brock = "gran"
	endif
		Name variable for Granodiorite

Whittle™	if (lith eq 6) then
	brock = "lptf"
	endif
		Name variable for Lapilli Tuff

Whittle™	if (lith eq 7) then
	brock = "qtzp"
	endif
		Name variable for Quartz Feldspar Porphyry

Whittle™	if (lith eq 8) then
	brock = "mfvl"
	endif
		Name variable for Mafic Metavolcanics

Whittle™	if (lith eq 9) then
	brock = "mets"
	endif
		Name variable for Metasediments

Whittle™	if (lith eq 10) then
	brock = "amps"
	endif
		Name variable for Amphibolite

Whittle™	if (lith eq 11) then
	brock = "dike"
	endif
		Name variable for Dikes

Pit Design	zone = 0 berm = 7.5 batter = 64	Default Slope Angles

Pit Design	if (brock eqs "sap" or brock eqs "sapr") then
	berm = 5
	batter = 37
	zone = 1
	endif
		Saprolite and Saprock Slope Angles

Production Schedule	haul_cut = "waste"	Default

Production Schedule	if (baui ge 0.7) then
	haul_cut = "ore"
	endif
		Internal CoG for oxide at US$1,200 gold price

Production Schedule	product = "waste"	Default

Production Schedule	if (baui gt 0.7 and baui le 1.0) then
	product = "lg1200"
	endif
		CoG for US$1,200 gold price

Production Schedule	if (baui gt 1.0 and baui le 2.1) then
	product = "mg800"
	endif
		CoG for US$800 gold price

Production Schedule	if (baui ge 2.1) then
	product = "hg400"
	endif
		CoG for US$400 gold price

Haul	total_cyc = ore_cyc + stock_cyc + waste_cyc total_dis = ore_dis + stock_dis + waste_dis	Combination of cycle times for display

Source: SRK, 2015

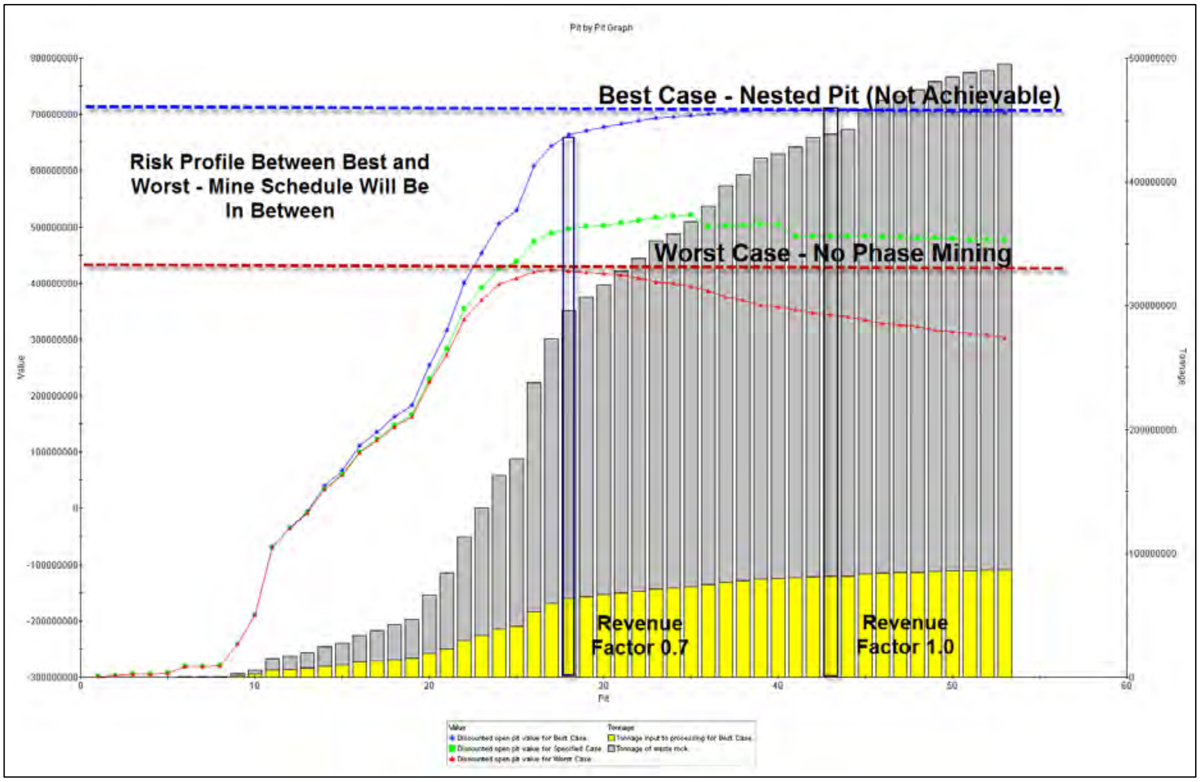
16.3 Pit Optimization

SRK used the Whittle™ pit optimization software to assist in determining the potential size of the resource to be considered in the PEA, production rates and phase sequencing for the PEA production schedule. Table 16.3.1 illustrates the SRK parameters used to analyze the potential Project size and to analyze the NPV potential of the deposit. The capital and operating costs were estimated at the time of the pit optimization and therefore will not necessarily match those later in the report.

Table 16.3.1: Pit Optimization Inputs

Whittle™ Parameter	Type	Common Parameters	Case 1	Case2	Case 3
Mining Cost	Reference Mining Cost (US$/t) Mining Cost Adjustment	$0.01/Bench Down $0.02/Bench Up	$ 1.50 $ 2.00 $ 2.50	$ 1.50 $ 2.00 $ 2.50	$ 1.50 $ 2.00 $ 2.50
Processing Cost Rock Type Process Cost (US$/crushed-t) Recoveries Grade	Process Name Selection Method Process Cost (US$/mill-t) Au Recovery - Saprolite (%) Au Recovery – Fresh (%) Lower cut-off	Mill Cut-off 0.9 0.9 Internal	$15.5 + $7	$14.757 + $4.57	$14.25 + $3.42
Revenue and Selling Cost	Au Price(US$/oz)	US$ 1,200			
Royalty, Refining, Transport etc.	Au Selling Cost (US$/oz)	$55.08 + $8			
Optimization	Revenue factor range	0.02-1.2 61 factors			
Operational Scenario – Time Costs	Initial Capital Cost Discount Rate Per Period	10%	$250 million	$300 million	$350 million
Operational Scenario – Limits	Process Limit (t/y)		3,600,000	5,475,000	7,300,000

Source: SRK, 2015



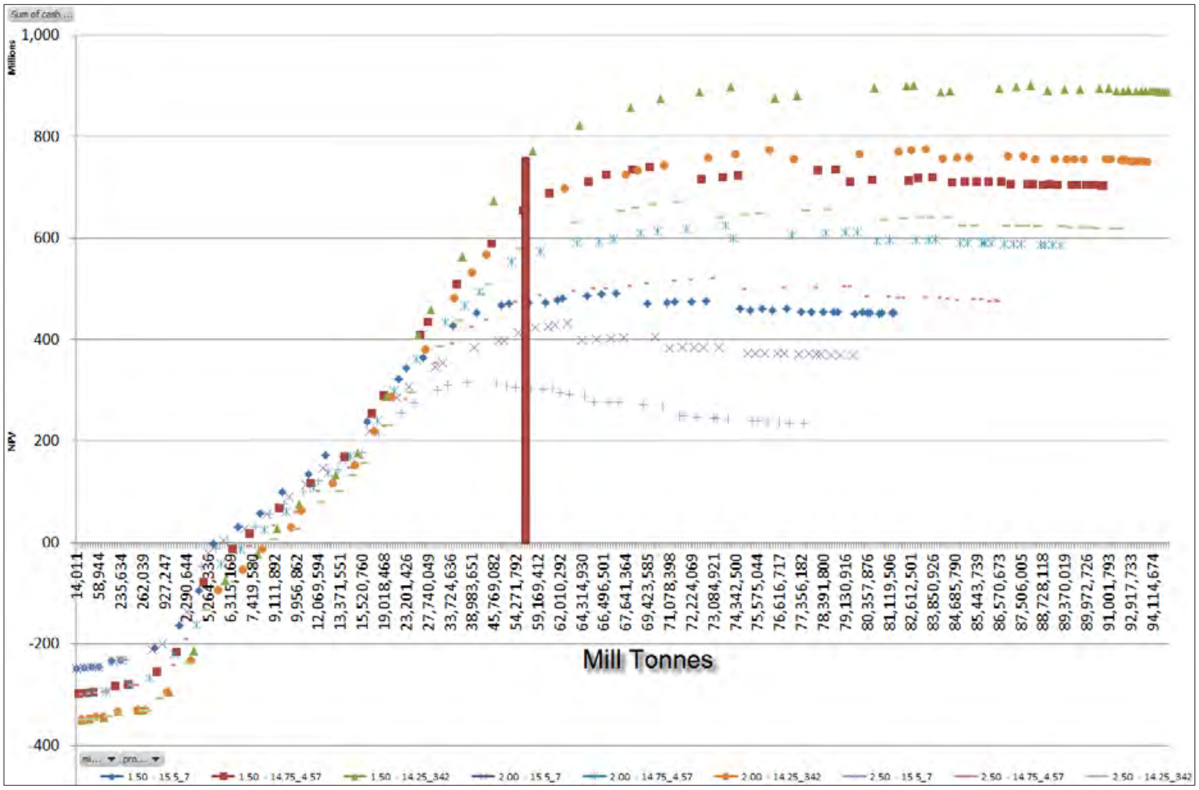
Source: SRK, 2015

Figure16.3.1: Pit by Pit Analysis

For Case 2 using US$2.50 mining scenario, the pit optimization graph shows that the pit is very sensitive to stripping ratio, given the resources defined in the model. As shown in Figure 16.3.1 the separation of the best (blue) and worst (red) discounted cashflow is minimal through pit 25. Between pit 25 and pit 28, there is the potential to make additional Project value, but the amount of stripping required for the extra resources starts to separate the best and worst case scenarios. After pit 28, very little additional value is added because the mill feed mined has such a high incremental cost in terms of stripping that no additional value is gleaned. Essentially, the mine is swapping ounces for tonnes mined. SRK targeted pit 28 or approximately 60 Mt for the basis of the pit design.

Because the Montagne d'Or deposit is large, but has a relatively high stripping ratio for the grade mined, the production rate becomes very important to determine Project value. To understand the effect of production rate, mine cost and effect of initial capital, a series of optimizations were run to determine the "value" spread of the different scenarios. This was then compared to the tonnage defined in the PEA pit design to confirm the tonnage was acceptable and robust.

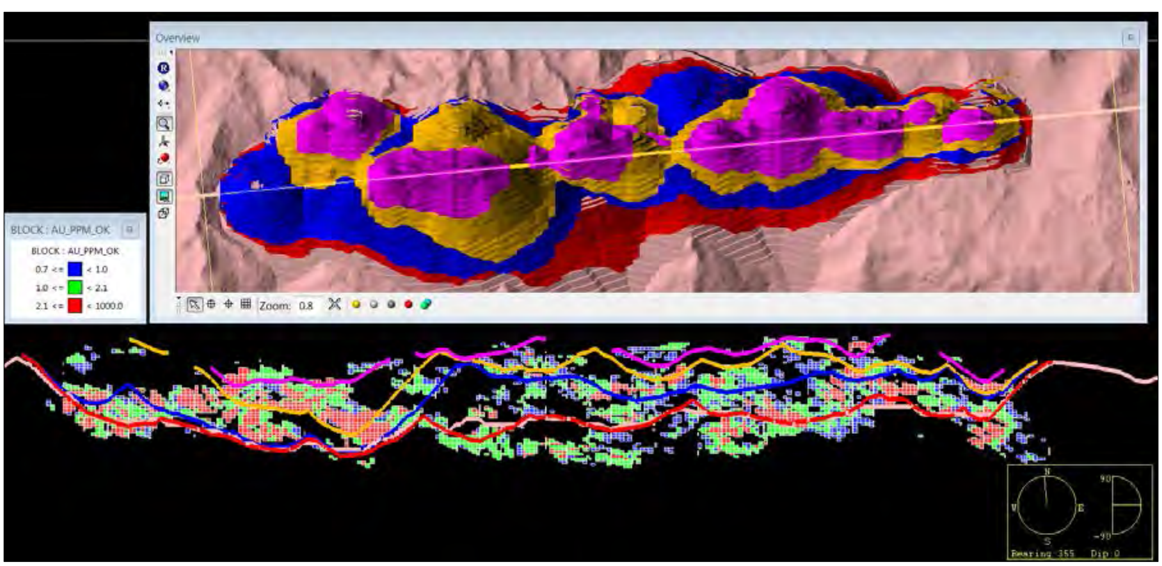
In Figure 16.3.2, the red line represents the resource tonnage defined in the PEA pit. Across all the different scenario cases, the majority of value is captured in the pit and in only the most conservative cases is there any value loss. It is also evident that in nearly all the scenarios, going beyond 60 Mt does not add additional value even with the extra throughput and lower mining costs of the optimistic scenarios. The selection was made from a discounted cashflow analysis and not IRR.



Source: SRK, 2015

Figure 16.3.2: Optimization Analysis

From a visual perspective, Figure 16.3.3 illustrates the pit shells in real world coordinates. Essentially, the hot spots, or high value pits, run from southwest to northeast moving from the center of the pits outward.



Source: SRK, 2015

Figure 16.3.3: Pit Shells Compared to Resource Potential

16.4 Geotechnical

The available geotechnical drillhole data (BD Montagne d'Or_CSV_2015-04-02), includes:

- RQD (19,991 drill runs, 48,411 m of core);
- Index hardness (IRS) (8,916 measurements);
- Oriented structure (3,985 measurements);
- Density (109 measurements);
- Fracture density and joint roughness coefficient (JRC) (12,785 measurements); and
- Structure (2,542).

Stereographic projections were reviewed based on the oriented structure data. The geologic solids were reviewed in Vulcan™. The saprolite typically has 0 RQD and a hardness of less than R2. The remaining hard rock units consist of volcanics including tuffs and diabase. Hard rock hardness ranged from R3 to R5 and RQD was typically 70% to 100%.

Based on this information rock mass rating (RMR) was estimated for the saprolite and the hard rock. Overall the structural data indicates a strong foliation that will govern angles in the footwall and hangingwall of the mine. Based on the available data, the estimated RMR for the tuffs and diabase ranges between 60 and 80. The estimated RMR for the saprolites is 20 to 30. No rock strength testing data was available.

Pit Design Concept

The pit will be mined in stages to allow mineral recovery along with waste removal. Acknowledging that these pit shells, and phases may change as additional data are collected and analyzed, they become the starting point for assessing appropriate geotechnical mine design parameters.

Stability of the pit is primarily dependent on the overall height of the pit and strength of the rock mass. The Montagne d'Or open pit is expected to have final wall heights up to 400 m in height.

Assumed Pit Slope Design Constraints

Assuming a 5 m model block height, it should be possible to mine a 15 m bench height as three 5 m flitches before placing a berm. In competent rock areas, there is the possibility to stack operating benches to a double benched configuration (30 m). The double bench configuration permits a steeper Inter-Ramp slope angle (IRA). Minimum bench width is assumed as follows:

- 7.5 m widths for single operating 15 m bench height; and
- 10.5 m widths double 30 m bench height.

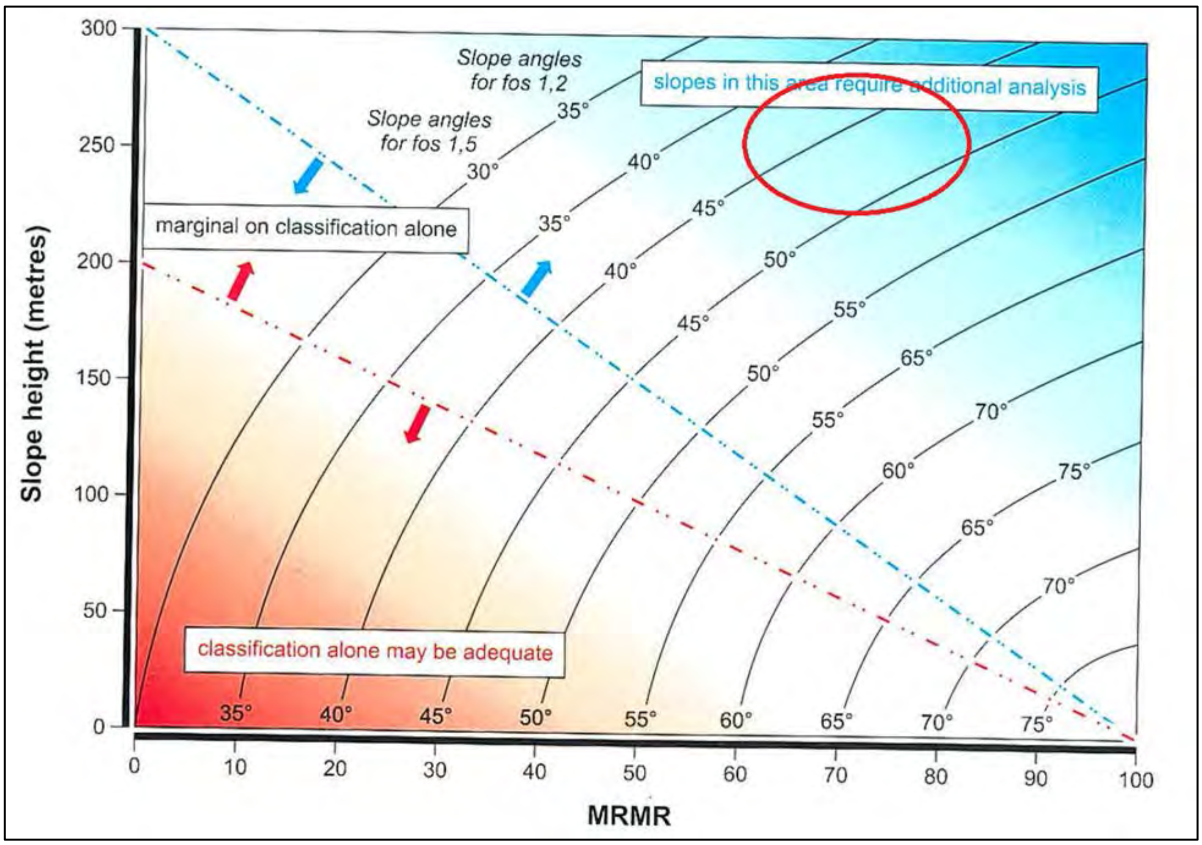
Average annual rainfall is in excess of 2,000 mm per year. Groundwater elevations are expected to be near the ground surface. It is anticipated that pit dewatering will be required to depressurize the rock mass below the pit walls. This will be necessary to maintain the required stability of pit walls. Dewatering will be on an inter-ramp scale on all rock walls. Surface water drainage will be required around the saprolites on the pit rim.

The minimum ramp width is 23 m for the type of haul trucks planned. The single bench is preferred for arresting ravel-type rockfalls and localized wedge instabilities, while a double bench may be used to increase the inter-ramp angles. Catch benches should include a 1 m berm at the crest to increase the effective catchment width for rockfall.

A geotechnical berm or ramp should be vertically spaced at a minimum of 150 m between catch bench stacks. The minimum width of the berm or ramp between bench stacks should be 20 m.

The overall pit angle required for stability is assumed to be a single consistent angle, below the saprolites. Insufficient geotechnical data exists to justify a compound pit angle. Maximum wall heights will control stability. The average pit height on the hangingwall side will be approximately 250 m. On the south or footwall side the maximum height will be a function of how far the saprolites will be mined but the slopes in hard rock will range from 350 to 400 m in height.

An empirical approach to assessing maximum overall pit angle is adopted for this study based on work by Haines and Terbrugger (1991). Figure 16.4.1 indicates that an overall pit angle of 50° in a rock mass with a Mining Rock Mass Rating (MRMR) range of 60 to 70 should have a FOS of about 1.3 for a pit wall height of 250 m. It is estimated that MRMR is about 90% of Rock Mass Rating (RMR) values after accounting for weathering, orientation of structures, mining induced stresses, blasting disturbance and residual water after slope depressurization.



Source: Adopted from Haines and Terbrugger, 1991

Figure 16.4.1: Empirical Design Chart for MRMR versus Slope Height

Saprolite

Saprolite and saprock thickness varies through the deposit. Thickness typically ranges from 20 to 40 m thick. For slope angles, these units are grouped together and will be referred to as saprolite. Saprolite slopes may be mined at 37°, however a 5 m wide "step-out" is recommended at the base of the saprock contact, and at each bench. Deformation and slope displacements should be expected at the proposed 37° slope angle. The step-out is recommended to maintain an adequate width for equipment to mine out failed saprolite material. A stable saprolite slope angle would be in the 18° to 24° range. Bench heights should be limited to 15 m in the saprolite.

16.4.1 Pit Design Parameters

Table 16.4.1.1 is a summary of the recommended geotechnical design parameters for the open pit design. As discussed above the inter-ramp slope angle in the saprolites will be 37°, with a 5 m wide berm in between benches.

Table 16.4.1.1: Recommended Pit Design Parameters (Hard Rock)

Parameter	Unit	Single Bench Design 15 m		Double Bench Design 30 m	
		Footwall Value	Hangingwall Value	Footwall Value	Hangingwall Value
Maximum Overall Slope Angle (OSA)	Degrees	45	41	49	45
Inter-ramp Slope Angle (IRA)	Degrees	49	45	54	59
Batter Face Angle (BFA)	Degrees	70	64	70	64
Bench Height	m	15	15	30	30
Berm Width	m	7.5	7.5	10.5	10.5

- Maximum inter-ramp slope height (bench stack height) is 150 m. A ramp or geotechnical berm with a width of 14 m is required between bench stacks.
- A minimum step-out of 5 m is required at the base of the saprolite/saprock.

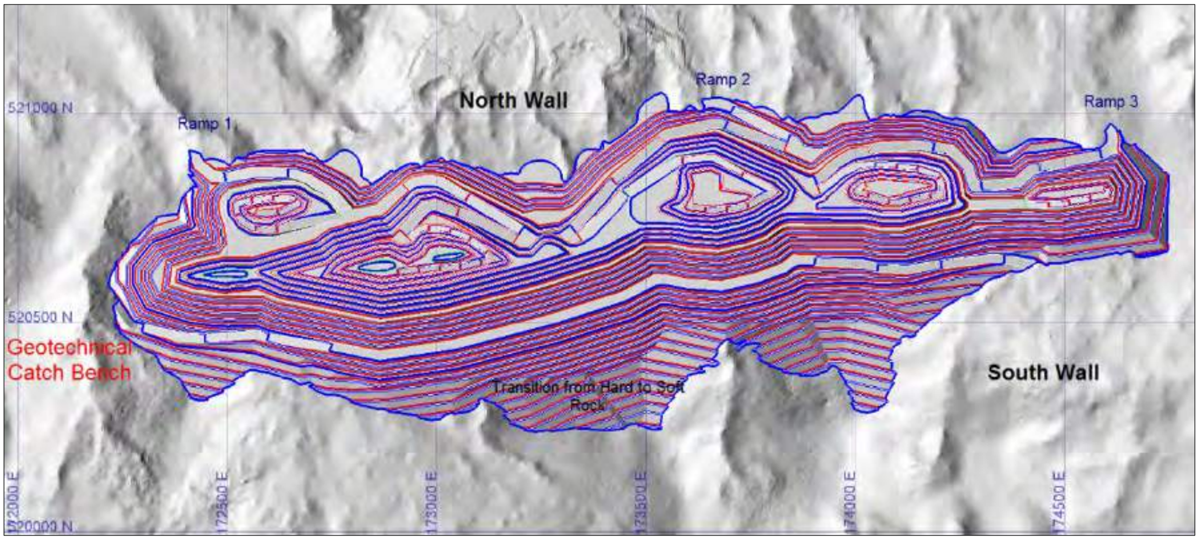
Source: SRK, 2015

16.5 Pit Design

The Montagne d'Or pit design is defined by a relatively high pit wall on the south side of the deposit that intersects hard and soft saprolite rock. The south wall is complicated by the requirement for geotechnical catch bench and the wall angle variation at the contact of hard and soft rock. The south wall toe location is a primary driver to the economics of the pit as the stripping ratio is highly sensitive to changes with the pit toe on this wall.

Ramps are placed in the north wall of the pit where stripping ratio penalty is relatively minor and three have been placed to ensure access to waste dumps and crusher access are not impeded.

Figure 16.5.1 illustrates the PEA pit design used in the evaluation of Montagne d'Or.



Source: SRK, 2015

Figure 16.5.1: PEA Pit

The pit design parameters applied are detailed in Table 16.5.1.

Table 16.5.1: Pit Design Parameters

Parameter	Unit	Saprolite	Fresh	Fresh Footwall	Dumps and Stockpiles
Overall Slope Angle	°	31	45	49	30
Batter Angle	°	37	64	70	37
Berm Placement Height	m	15	15	15	25
Flitch (Mining Face) Height	m	5	5	5	25
Berm Width	m	5	7.5	7.5	10
Catch Bench Berm Width	m	None	23	None	NA
Max Height to Catch Bench	m	None	Approx 185 m	None	NA
Ramp Width – 2 way	m	23	23	23	23
Ramp Width – 1 way	m	14	14	14	NA
Ramp Gradient (Shortest)	%	10	10	10	10

Source: SRK, 2015

The ramp width of 23 m for CAT 777 trucks is reasonable. The ramp to truck width is 3.3 assuming a 6.9 m operating width. Ramps are limited to a 10% grade based on the shortest ramp distance; this means that ramps should never exceed 10% in the pit design. Single lane roads of 14 m for a CAT 777 are generally limited to 3 benches at the pit bottom. Given a mixed fleet with ADT's it would be possible to make the ramps even smaller with further iterations of the pit design.

Waste dumps are placed in 25 m lifts with a 10 m berm to reduce the velocity of water running on the dump face thus reducing problems dump face erosion from the heavy rainfall events in the region, and to improve overall geotechnical stability.

The footwall or north side of the pit wall has a slope angle of 50° as the berm width is reduced to 7 m instead of 7.5 m.

The Vulcan™ block model was flagged with fresh and saprolite geotechnical conditions to control the transition from fresh rock to saprolite wall angles as the pit steps up through the transition of the two rock types.

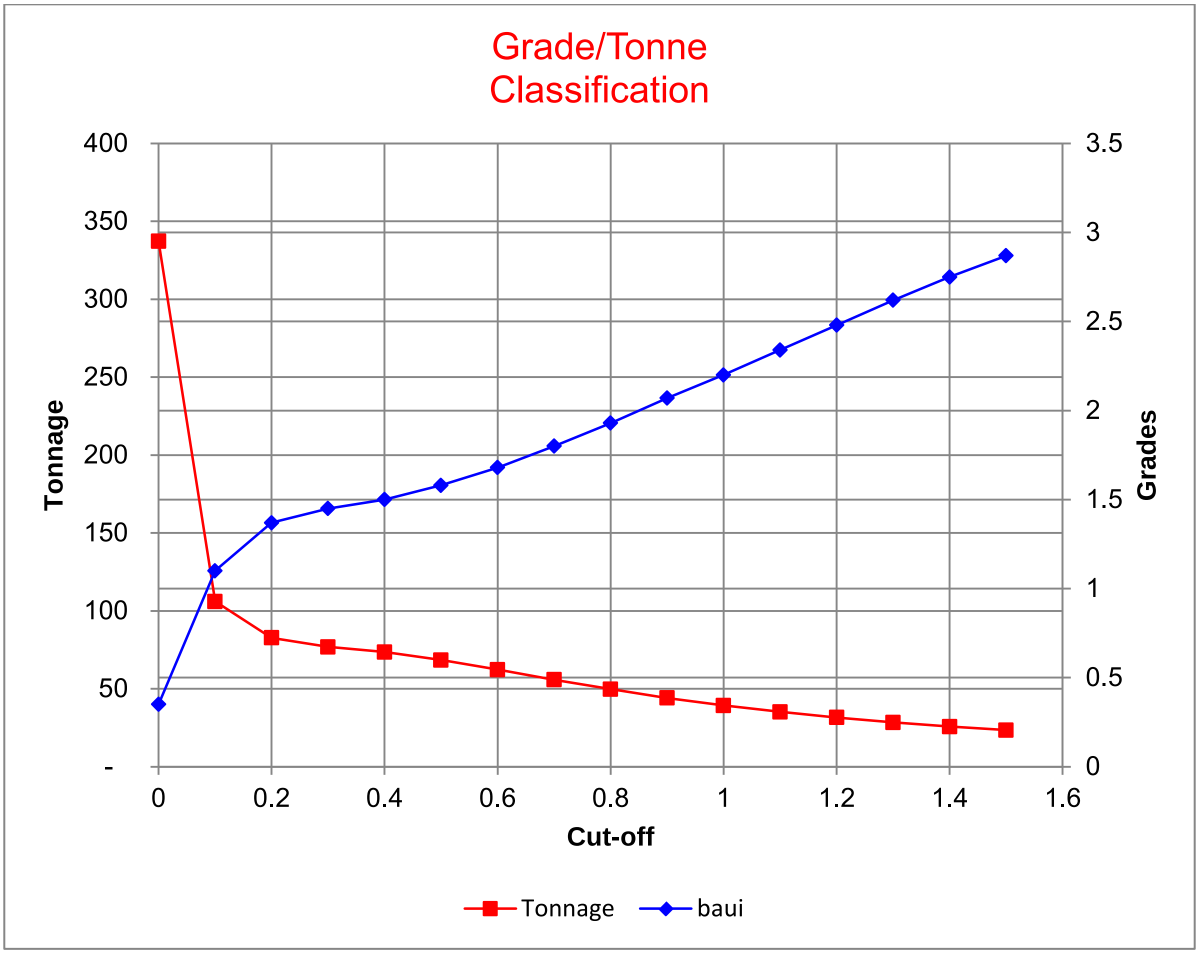
16.6 Grade Tonnage

The grade tonnage for the PEA pit is detailed in Table 16.6.1 and is visually displayed in Figure 16.6.1.

Table 16.6.1: Grade Tonnage Curve within the PEA Pit Design

Cut-off	Au	Tonnage (Mt)	Ounces (Moz)	Stripping Ratio (W:O)
0	0.35	337	3.80	-
0.1	1.10	106	3.75	2.18
0.2	1.37	83	3.64	3.08
0.3	1.45	77	3.58	3.39
0.4	1.50	74	3.55	3.59
0.5	1.58	68	3.48	3.93
0.6	1.68	62	3.37	4.41
0.7	**1.80**	**56**	**3.23**	**5.04**
0.8	1.93	50	3.09	5.78
0.9	2.07	44	2.93	6.65
1.0	2.20	39	2.78	7.58
1.1	2.34	35	2.65	8.58
1.2	2.48	32	2.51	9.69
1.3	2.62	28	2.39	10.90
1.4	2.75	26	2.27	12.11
1.5	2.87	23	2.17	13.37

Source: SRK, 2015

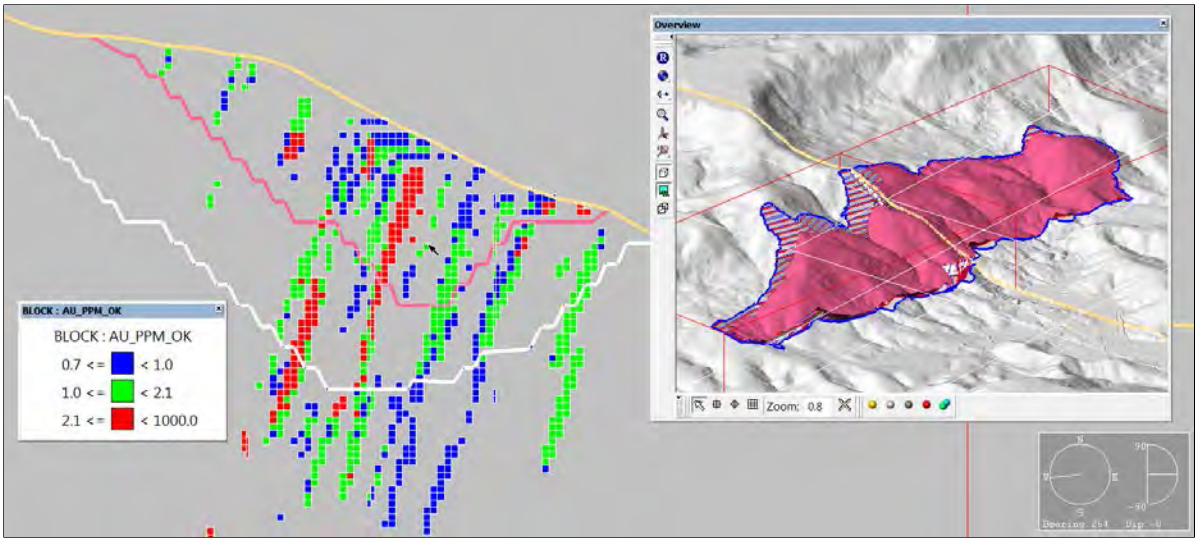


Source: SRK, 2015

Figure 16.6.1: Grade Tonnage Curve within PEA Pit

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The PEA pit design encompasses a wide range of possible production scenarios, provided over 3 Moz with a strip ratio less than 6:1. SRK also completed a pit design that targeted the best IRR rather than NPV. The second pit design grade tonnage is displayed in Table 16.6.2 and an example of the pit comparisons in Figure 16.6.2.



Source: SRK, 2015

Figure 16.6.2: Grade Tonnage Curve within Maximum IRR Pit

The majority of the variation is to the eastern part of the open pit. The west side has very little geometrical difference between the PEA pit and the Maximum IRR pit. The IRR pit was developed fully for comparison with the selected PEA pit shell but may be considered in the FS depending on corporate strategy for the deposit.

Table 16.6.2: Grade Tonnage Curve within the Maximum IRR Pit Design

Cut-off	Au	Tonnage (Mt)	Ounces (Moz)	Stripping Ratio (W:O)
-	0.37	227	2.70	-
0.10	1.09	76	2.66	1.99
0.20	1.40	57	2.57	2.97
0.30	1.50	53	2.56	3.29
0.40	1.55	51	2.52	3.49
0.50	1.64	47	2.47	3.84
0.60	1.74	43	2.39	4.31
0.70	**1.87**	**38**	**2.31**	**4.92**
0.80	2.00	34	2.21	5.61
0.90	2.14	31	2.11	6.41
1.00	2.28	27	2.01	7.28
1.10	2.41	25	1.92	8.18
1.20	2.55	22	1.83	9.16
1.30	2.67	20	1.74	10.18
1.40	2.79	19	1.67	11.19
1.50	2.91	17	1.60	12.27

Source: SRK, 2015

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

16.7 Cut-Off Grade

SRK selected three CoGs that represent a US$1,200/oz gold price for calculating resources in the Montagne d'Or mine plan, another at US$800/oz for mid-grade and another at US$400/oz representing high grade. A breakeven CoG was used rather than the internal CoG and estimates were made before the final economic model was created.

Table 16.7.1 details the internal CoG for high grade, mid-grade and low grade.

Table 16.7.1: Cut-Off Calculations and Grade Bins

Description	Units	LG 1200 CoG	MG 800 CoG	HG 400 CoG
Assumptions				
Gold Price	US$/oz	$1,200	$800	$400
Gold Price	US$/g	$38.58	$25.72	$12.86
Smelting & Refining	US$/oz	$8	$8	$8
Royalty (NSR)	%	4.59%	4.59%	4.59%
Au Grade	g/t	0.69	1.04	2.13
Au Recovery	%	90.0%	90.0%	90.0%
Operating Costs				
Smelting & Refining	US$/t milled	$0.00	$0.00	$0.00
Royalty	US$/t milled	$1.11	$1.11	$1.13
Mining	US$/t mined	$2.50	$2.50	$2.50
Processing	US$/t milled	$20.00	$20.00	$20.00
HL Pad Costs	US$/t milled	$0.50	$0.50	$0.50
Other Costs (e.g. Reclamation)	US$/t milled	$0.00	$0.00	$0.00
G&A	US$/t milled	$0.00	$0.00	$0.00
Subtotal	US$/t milled	**$24.11**	**$24.11**	**$24.11**
CoG - Head Grade	g/t	0.69	1.04	2.13
CoG - Recovered Grade	g/t	0.62	0.94	1.91
Value Used In Scripts	g/t	0.7	1.0	2.1

Source: SRK, 2015

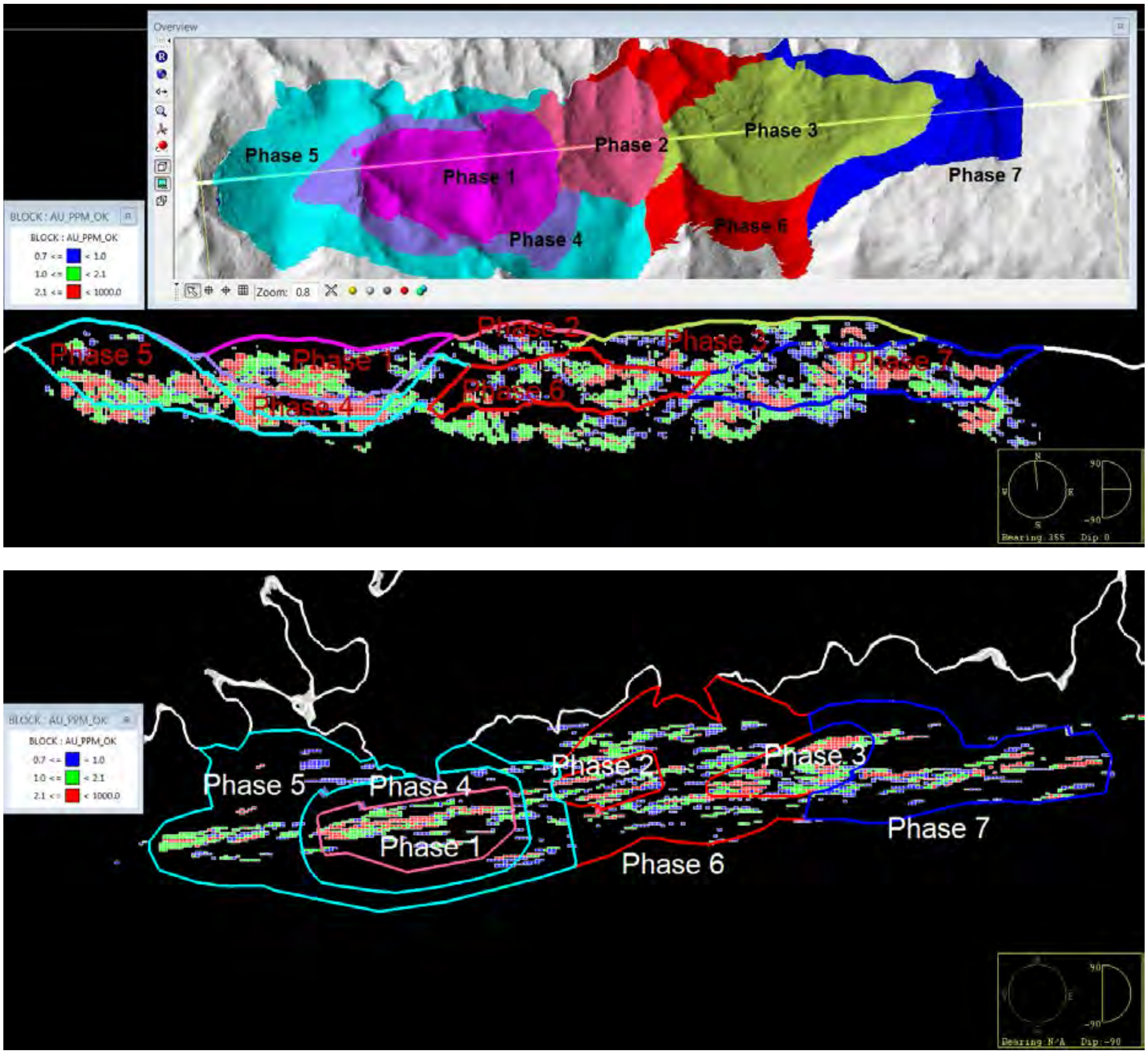
16.8 Phase Design

Phase designs were constructed by splitting up the final PEA pit into smaller and more manageable pieces while still ensuring each bench within each phase has ramp access. The phases have been developed by balancing mining constraints with the optimum extraction sequence suggested by pit optimization results presented previously in Section 16.3, Figure 16.3.2.

Some of the basic design parameters include:

- Target low strip ratio pit along strike of the deposit;
- Excavate western section of the pit first before mining lower grade deep eastern mill feed;
- Attempt to keep the minimum mining width at approximately 50 to 100 m; and
- Same pit parameters for the PEA pit design have been used for phase design.

Figure 16.8.1 shows a graphical representation of the seven phases that were constructed for the LoM production schedule inventory.



Source: SRK, 2015

Figure 16.8.1: Phase Layout

Once the phases have been designed, solid triangulations are created for each phase as it cuts into topography from previous phases. These solid phases are then shelled (cut) on a 15 m lift height that corresponds to three block model blocks. These shells form a bench within each phase and represent the basic unit that is scheduled for the LoM production plan.

Table 16.8.1 illustrates the phase inventory that has been used as the basis for the LoM production schedule.

Table 16.8.1: Phase Inventory for Production Schedule

	Gold Grade (g/t)	Tonnes (000's)	Gold Grade (g/t)							Tonnes (000's)						
Phase	**Total**	**Total**	**P1**	**P2**	**P3**	**P4**	**P5**	**P6**	**P7**	**P1**	**P2**	**P3**	**P4**	**P5**	**P6**	**P7**
MILL	**1.80**	**55,807**	**2.06**	**1.60**	**1.80**	**2.10**	**1.99**	**1.56**	**1.75**	**5,328**	**4,051**	**8,671**	**3,709**	**10,956**	**13,589**	**9,504**
FLPY	1.03	91		0.96	1.12		1.11	0.98	1.05		11	4		27	36	13
FLTF	1.89	42,590	2.26	1.81	1.91	2.21	2.11	1.59	1.72	4,290	2,707	6,526	3,293	9,015	12,060	4,699
GRAN	1.46	2,297	1.07	1.51	1.79	1.33	1.65	1.08		282	99	59	346	1,228	282	
LPTF	1.14	58					1.14							58		
MFVL	1.76	6,380			1.74			1.49	1.82			881			963	4,536
QTZP	1.58	71	0.78	0.95		0.81	2.03	0.76	1.67	1	4		9	16	1	41
SAP	1.17	2,875	1.29	1.17	1.20	1.05	1.05	1.05	1.14	534	841	718	45	416	180	140
SAPR	1.23	1,445	1.35	1.11	1.35	0.81	1.14	1.08	1.17	220	389	481	15	198	67	75
WASTE		**281,498**	**0.17**	**0.25**	**0.17**	**0.15**	**0.16**	**0.22**	**0.15**	**23,744**	**9,912**	**22,654**	**25,643**	**86,830**	**61,754**	**50,961**
DIKE		15,405								1,123	499	1,295	1,531	4,248	5,672	1,036
FLPY	0.12	4,910	0.03	0.20	0.11	0.03	0.07	0.15	0.15	96	114	412	413	1,087	1,027	1,761
FLTF	0.26	124,047	0.22	0.33	0.29	0.21	0.26	0.29	0.23	11,990	4,546	8,560	11,369	34,061	33,110	20,413
GRAN	0.12	32,559	0.18	0.18	0.07	0.12	0.10	0.14	0.07	2,498	639	581	4,730	16,293	6,945	873
LPTF	0.09	7,251	0.10			0.10	0.08			554			2,312	4,385		
METS		6,400								0			475	5,781	144	
MFVL	0.14	28,396			0.12			0.17	0.13			3,446			5,617	19,333
QTZP	0.15	3,883	0.10	0.32	0.08	0.13	0.08	0.24	0.14	341	85	112	448	1,295	1,020	583
SAP	0.13	44,624	0.12	0.20	0.12	0.12	0.14	0.11	0.07	5,491	2,913	6,035	3,237	15,596	6,379	4,973
SAPR	0.14	14,020	0.14	0.24	0.15	0.10	0.12	0.10	0.10	1,650	1,116	2,213	1,129	4,084	1,840	1,988
Total		**337,306**	**0.70**	**0.79**	**0.84**	**0.66**	**0.73**	**0.78**	**0.68**	**29,071**	**13,963**	**31,324**	**29,353**	**97,786**	**75,343**	**60,465**

Source: SRK, 2015

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

16.9 Production Schedule

The mine production schedule utilized the CPLEX optimization tool within the Maptek™ Chronos scheduling package. Benches within each phase have a precedence relationship assigned to ensure top down mining in an orderly sequence. This sequence was optimized to meet the constraints detailed in Table 16.9.1. The objective function of the optimization was to maximize the ounces of gold in each period but still maintain a reasonable mining fleet. To estimate the fleet requirements, the "Haul_Units" variable controlled the weighted tonnage that a mine fleet could excavate. Optimizations were conducted on four year intervals and involved trial and error to determine suitable upper and lower bounds for each period mined.

Table 16.9.1: CPLEX Optimization Limits

Variable	Mill Tonnes		Ounces		Waste		Haul Units	
Limit	Free		Maximize		Free		Free	
Year	Lower bound	Upper bound	Lower bound	Upper bound	Lower bound	Upper bound	Lower bound	Upper bound
(1)	450,000	750,000	-	-	-	-	-	-
1	4,500,000	5,500,000	315,000	330,000	-	25,000,000	300,000,000	450,000,000
2	4,500,000	5,500,000	315,000	330,000	-	25,000,000	300,000,000	450,000,000
3	4,500,000	5,500,000	315,000	330,000	-	35,000,000	300,000,000	450,000,000
4	4,500,000	6,500,000	315,000	330,000	-	35,000,000	300,000,000	600,000,000
5	4,500,000	6,500,000	275,000	330,000	-	35,000,000	300,000,000	600,000,000
6	4,500,000	6,500,000	275,000	330,000	-	35,000,000	300,000,000	600,000,000
7	4,500,000	6,500,000	275,000	330,000	-	35,000,000	300,000,000	600,000,000
8	4,500,000	6,500,000	275,000	330,000	-	35,000,000	300,000,000	600,000,000
9	4,500,000	5,500,000	275,000	330,000	-	35,000,000	300,000,000	600,000,000
10	4,500,000	5,500,000	275,000	330,000	-	35,000,000	300,000,000	600,000,000
11	-	5,500,000		330,000	-	35,000,000		600,000,000

Source: SRK, 2015

The annual mine production schedule for material coming directly from the pit, being stockpiled and then fed to the mill is detailed in Table 16.9.2.

In all periods (except pre-production) High-Grade (HG) and Medium-Grade (MG) are sent directly to the mill so only Low-Grade (LG) is stored in a stockpile for use at the end of the operational mine life.

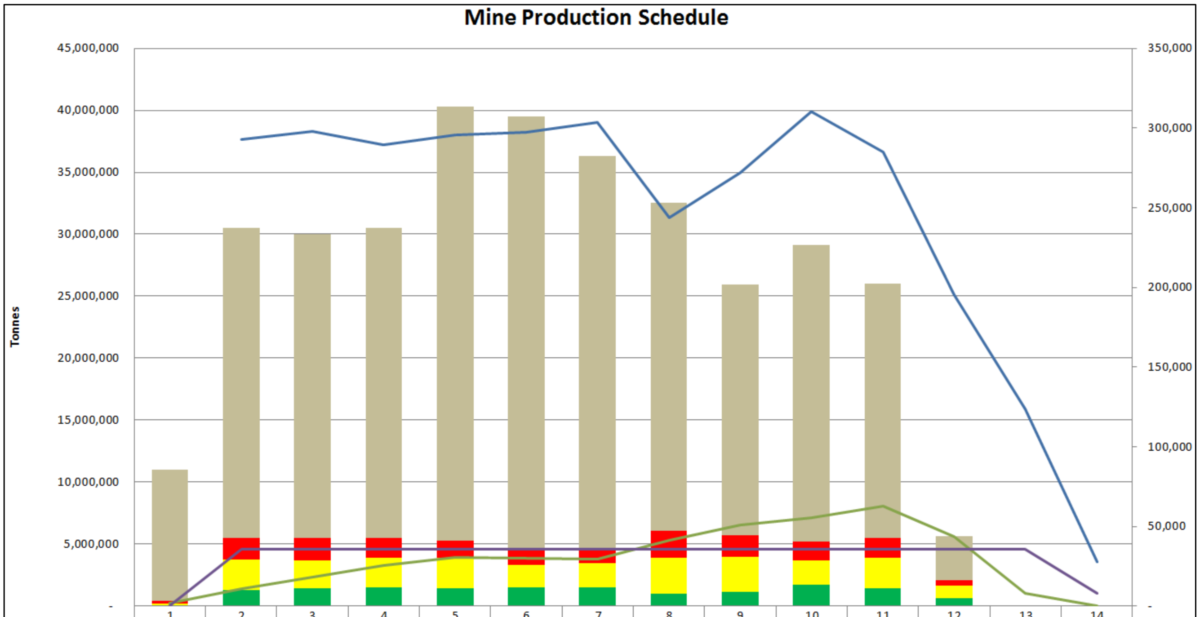
Table 16.9.2: Stockpile Inventory Schedule

	Units	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Mill Tonnes	t	**55,807**	450	5,500	5,500	5,500	5,276	4,500	4,500	6,079	5,739	5,186	5,500	2,078	0	
Mill Grade	g/t	**1.80**	1.04	1.78	1.83	1.78	1.86	2.04	2.09	1.46	1.65	1.96	1.75	1.92	-	
Ounces	oz	**3,234**	15	315	323	315	315	295	302	285	304	327	310	128	0	
Waste	t	**281,498**	10,550	25,000	24,504	24,982	35,000	35,000	31,799	26,438	20,209	23,934	20,523	3,560	0	
Total Tonnes	**t**	**337,306**	**11,000**	**30,500**	**30,004**	**30,482**	**40,276**	**39,500**	**36,299**	**32,517**	**25,948**	**29,120**	**26,023**	**5,637**	**0**	
HG400 RoM	t		3	1,275	1,450	1,506	1,455	1,479	1,538	970	1,136	1,707	1,459	658	0	
HG400 RoM Au	g/t		2.60	3.77	3.68	3.36	3.52	3.73	3.72	3.12	3.52	3.60	3.28	3.31	-	
HG400 RoM to Plant	t		0	1,275	1,450	1,506	1,455	1,479	1,538	970	1,136	1,707	1,459	658	0	
HG400 Stock Begin	t		0	3	0	0	0	0	0	0	0	0	0	0	0	
MG800 RoM	t		210	2,511	2,232	2,377	2,441	1,847	1,948	2,923	2,839	1,939	2,461	959	0	
MG800 RoM Au	g/t		1.23	1.41	1.43	1.41	1.44	1.45	1.44	1.36	1.40	1.40	1.43	1.47	-	
MG800 RoM to Plant	t		0	2,511	2,232	2,377	2,441	1,847	1,948	2,923	2,839	1,939	2,461	959	0	
MG800 Stock Begin	t		0	210	0	0	0	0	0	0	0	0	0	0	0	
LG1200 RoM	t		237	1,715	1,818	1,617	1,380	1,173	1,014	2,185	1,764	1,540	1,581	461	0	
LG1200 RoM Au	g/t		0.85	0.85	0.84	0.85	0.84	0.83	0.84	0.85	0.85	0.84	0.84	0.84	-	
LG1200 RoM to Plant	t		0	577	881	680	666	1,173	1,014	669	588	929	643	461	0	
LG1200 Stock Begin	t		0	237	1,375	2,313	3,250	3,964	3,889	3,826	5,343	6,519	7,130	8,067	5,582	1,020
LG1200 Stock In	t		237	1,138	938	937	714	0	0	1,516	1,176	611	938	0	0	0
LG1200 Stock Out	t		0	0	0	0	0	75	62	0	0	0	0	2,485	4,563	1,020
LG1200 Stock End	t		237	1,375	2,313	3,250	3,964	3,889	3,826	5,343	6,519	7,130	8,067	5,582	1,020	0
LG1200 Stock Begin Au	g/t		-	0.85	0.85	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	
LG1200 Stock In Au	g/t		0.85	0.85	0.84	0.85	0.84	-	-	0.85	0.85	0.84	0.84	-	-	0.84
LG1200 Stock Out Au	g/t		-	-	-	-	-	0.84	0.84	-	-	-	-	0.84	0.84	-
LG1200 Stock End Au	g/t		0.85	0.85	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84
p1_benches	number		6.00	4.72	3.29	0.98	1.00	-	-	-	-	-	-	-	-	-
p2_benches	number		6.02	4.56	1.42	1.64	1.36	-	-	-	-	-	-	-	-	
p3_benches	number		4.17	0.83	4.36	3.64	4.00	-	-	-	-	-	-	-	-	
p4_benches	number		-	6.00	2.00	4.00	7.99	3.01	-	-	-	-	-	-	-	
p5_benches	number		-	9.15	2.70	2.13	2.63	4.16	5.27	0.96	-	3.00	-	-	-	
p6_benches	number		-	-	-	-	2.04	8.96	4.92	5.76	3.73	4.60	-	-	-	
p7-Benches	number		-	-	-	-	-	6.86	-	-	3.14	4.21	5.13	4.66	-	
Plant Feed	t	**55,807**	0	4,575	4,563	4,563	4,563	4,575	4,563	4,563	4,563	4,575	4,563	4,563	4,563	1,020
Plant Feed Au	g/t	**1.80**	-	1.99	2.03	1.97	2.02	2.02	2.07	1.66	1.86	2.11	1.94	1.33	0.84	0.84
Ounces	oz	**3,234**		293	298	290	296	297	303	244	272	310	285	195	124	28
Stock Re-Handle	t	**8,418**	0	213	0	0	0	75	62	0	0	0	0	2,485	4,563	1,020

Source: SRK, 2015

The highest bench sinking rate occurs in year 5 because as a stripping hurdle must be overcome, and even so there is a drop in available mill feed in year 6. The multiple benches mined from each phase in each year will provide good operational flexibility for the mine.

Figure 16.9.1 illustrates the annual production schedule. The pre-strip is targeted to supply the necessary fill for tailings dam construction and other earthworks to be determined in the future. The proportion of HG, MG and LG is reasonably consistent through the mine life, with only a lean period in years 6 and 7, while the final phases are mined to the pit bottom on the west side of the pit. That is evident by the drop in total grade. More detailed phase design may provide the necessary flexibility to avoid this in the future.

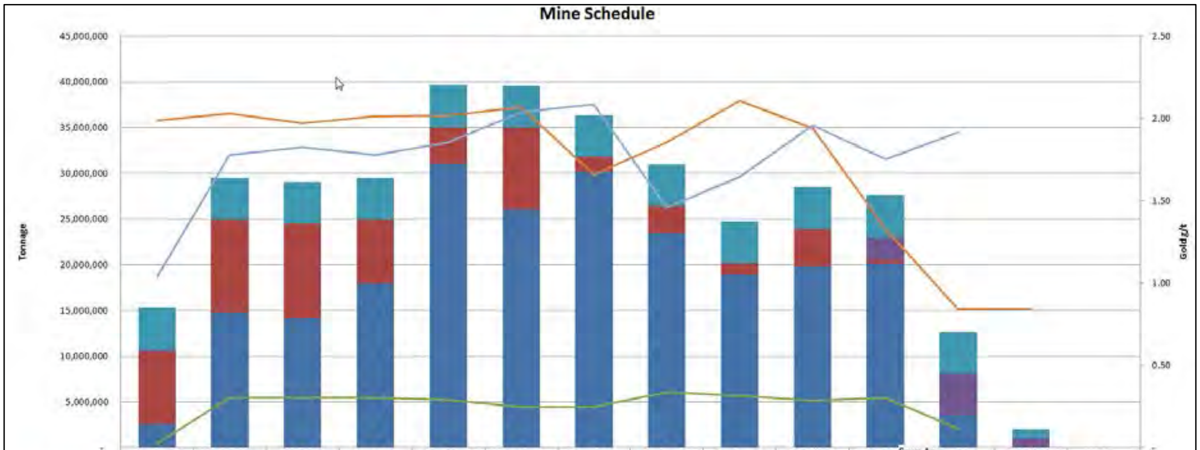


	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Waste	10,550,00	25,000,00	24,503,64	24,981,84	35,000,00	35,000,00	31,799,08	26,438,09	20,209,00	23,933,69	20,523,04	3,559,789		
LG Tonnes	237,051	1,714,734	1,818,372	1,617,337	1,380,166	1,173,414	1,013,796	2,184,973	1,764,468	1,539,769	1,580,684	460,975		
MG Tonnes	209,760	2,510,635	2,231,756	2,376,727	2,440,960	1,847,411	1,948,490	2,923,365	2,838,651	1,938,744	2,460,540	958,572		
HG Tonnes	3,189	1,274,632	1,449,873	1,505,936	1,455,241	1,479,175	1,537,714	970,369	1,135,571	1,707,499	1,458,776	658,022		
LG Stock	237,051	1,375,000	2,312,500	3,250,000	3,963,867	3,888,867	3,826,367	5,342,573	6,518,763	7,129,775	8,067,275	5,582,344	1,019,844	-
Plant Feed	-	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	1,019,844
Mill Au Ounces		292,544	297,857	289,522	295,765	297,068	303,451	243,743	272,139	310,100	284,747	195,375	123,685	27,647

Source: SRK, 2015

Figure 16.9.1: Grade Bin Schedule

Figure 16.9.2 illustrates the proportion of saprolite versus hard rock in the production schedule. It also shows the mined gold grade versus the grade fed to the process plant. The steps correspond to the schedule defined in Table 16.9.1.



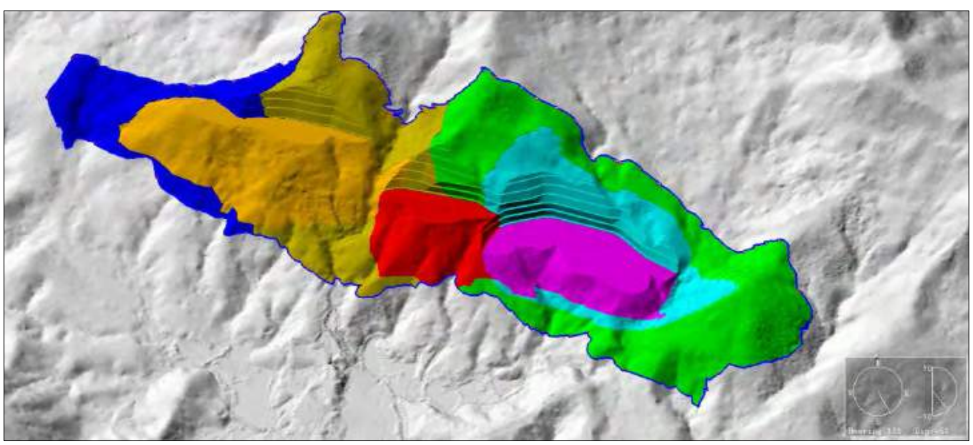
	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Plant Feed	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	1,019,844	
Rehandle	212,949	-	-	-	75,000	62,500	-	-	-	-	2,484,931	4,562,500	1,019,844	
Waste Saprolite	7,958,204	10,156,603	10,353,370	7,035,209	3,918,008	8,904,764	1,571,898	2,941,306	1,323,676	4,110,951	370,556	-	-	-
Waste Tonnes	2,591,796	14,843,397	14,150,278	17,946,638	31,081,992	26,095,236	30,227,190	23,496,786	18,885,329	19,822,745	20,152,486	3,559,789	-	-
Rom Mill Tonnes	450,000	5,500,000	5,500,000	5,500,000	5,276,367	4,500,000	4,500,000	6,078,707	5,738,690	5,186,012	5,500,000	2,077,569		
Plant Feed Au	1.99	2.03	1.97	2.02	2.02	2.07	1.66	1.86	2.11	1.94	1.33	0.84	0.84	
Rom Mill Grade	1.04	1.78	1.83	1.78	1.86	2.04	2.09	1.46	1.65	1.96	1.75	1.92		

Source: SRK, 2015

Figure 16.9.2: Saprolite, Hard Rock and RoM Production Schedule

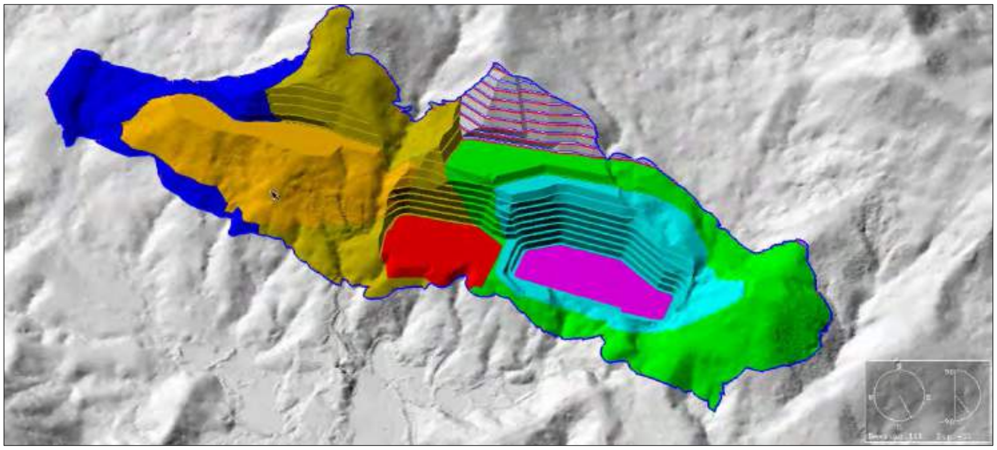
The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Figures 16.9.3 through 16.9.13 show the annual mine production sequence that corresponds to the production schedule detailed above. Due to the nature of the study pioneering roads have not been included in the phase design. During detailed mine planning ramps will need to be included on the high wall side of each phase.



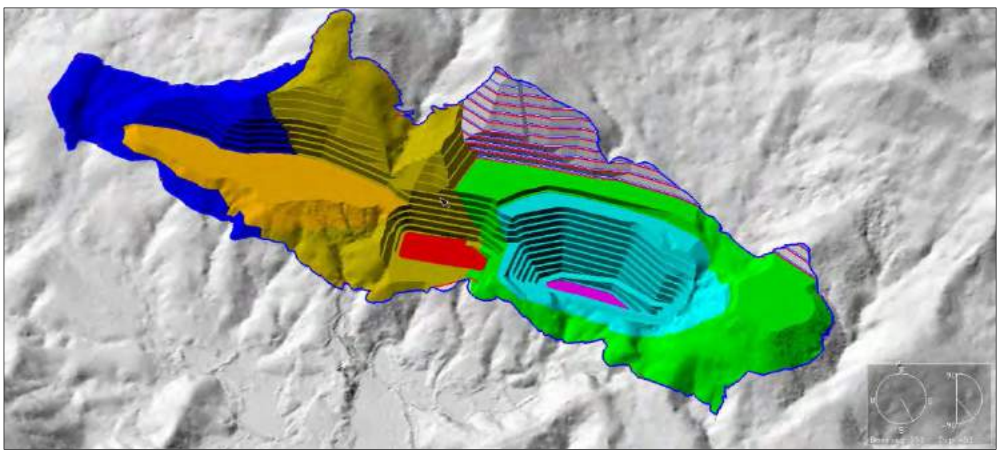
Source: SRK, 2015

Figure 16.9.3: Pre-Production Stripping



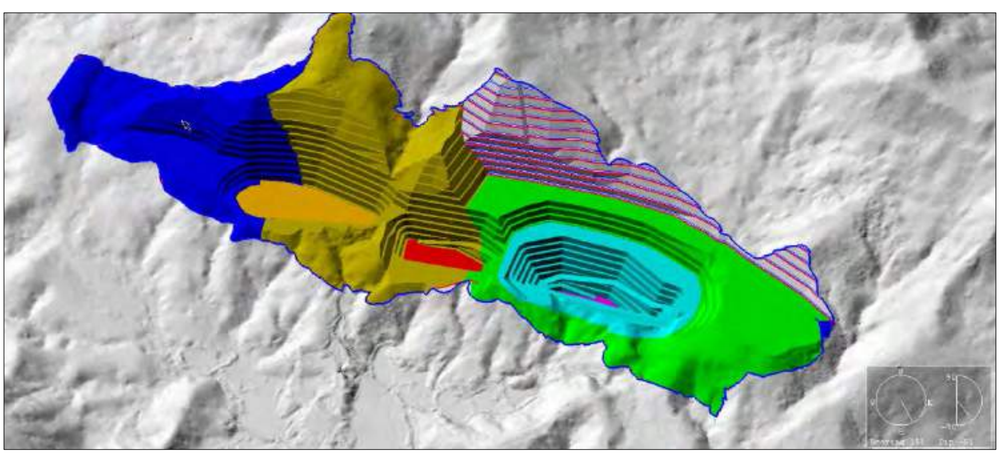
Source: SRK, 2015

Figure 16.9.4: Year 1 Pit Phase Plot



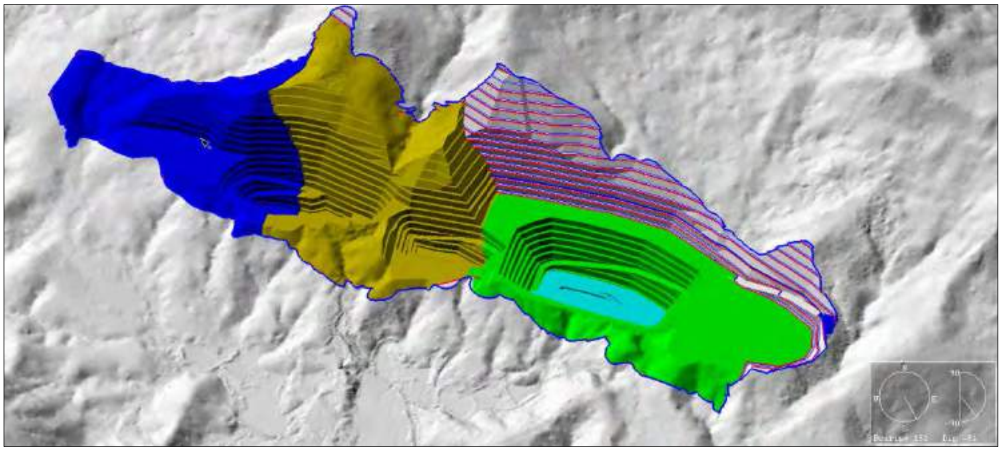
Source: SRK, 2015

Figure 16.9.5: Year 2 Pit Phase Plot



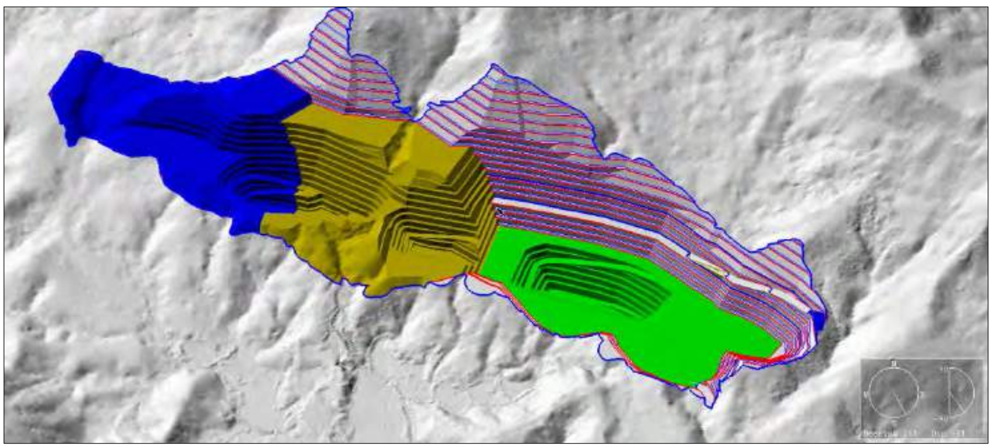
Source: SRK, 2015

Figure 16.9.6: Year 3 Pit Phase Plot



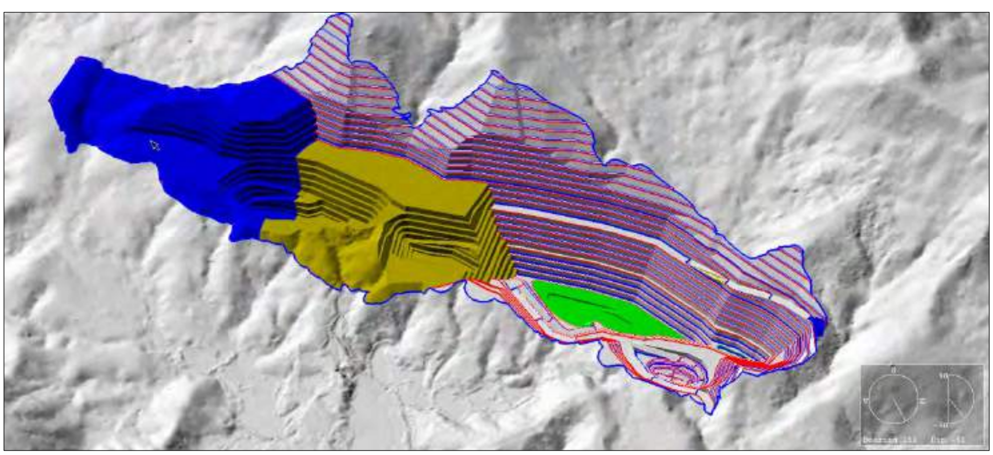
Source: SRK, 2015

Figure 16.9.7: Year 4 Pit Phase Plot



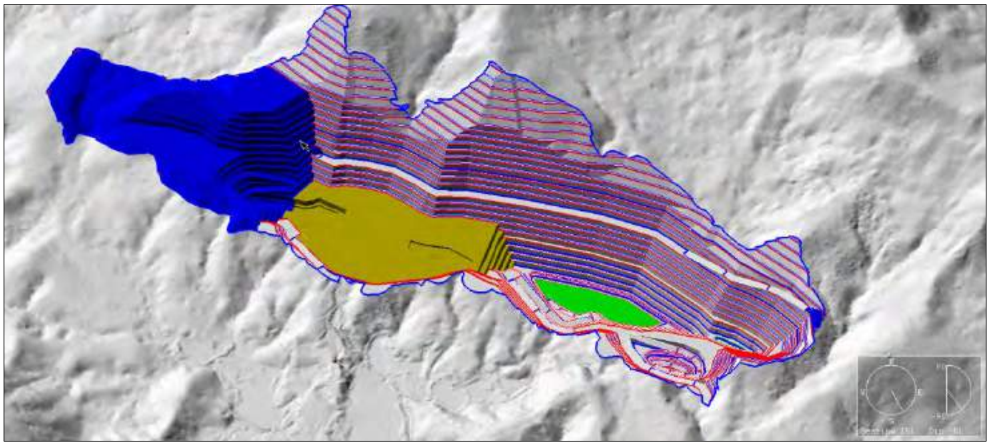
Source: SRK, 2015

Figure 16.9.8: Year 5 Pit Phase Plot



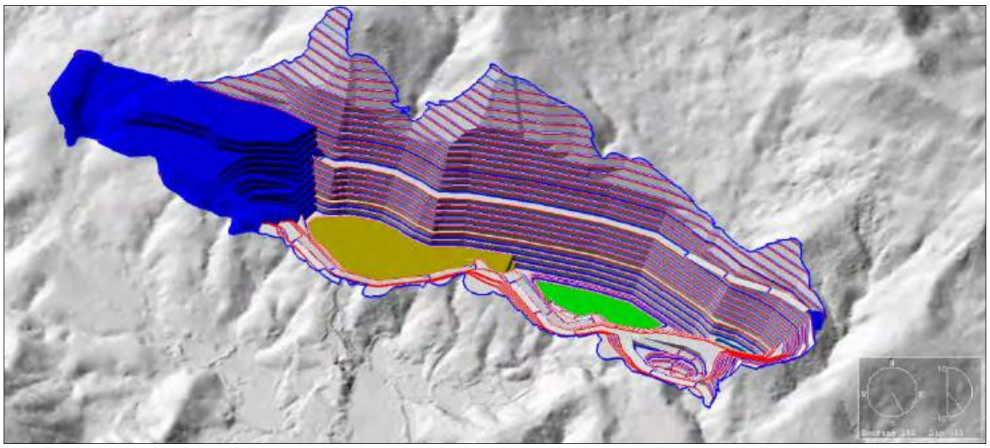
Source: SRK, 2015

Figure 16.9.9: Year 6 Pit Phase Plot



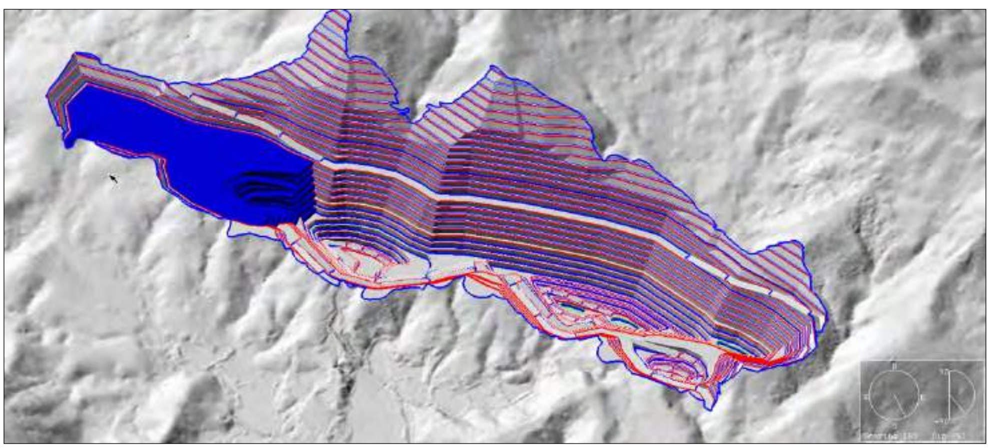
Source: SRK, 2015

Figure 16.9.10: Year 7 Pit Phase Plot



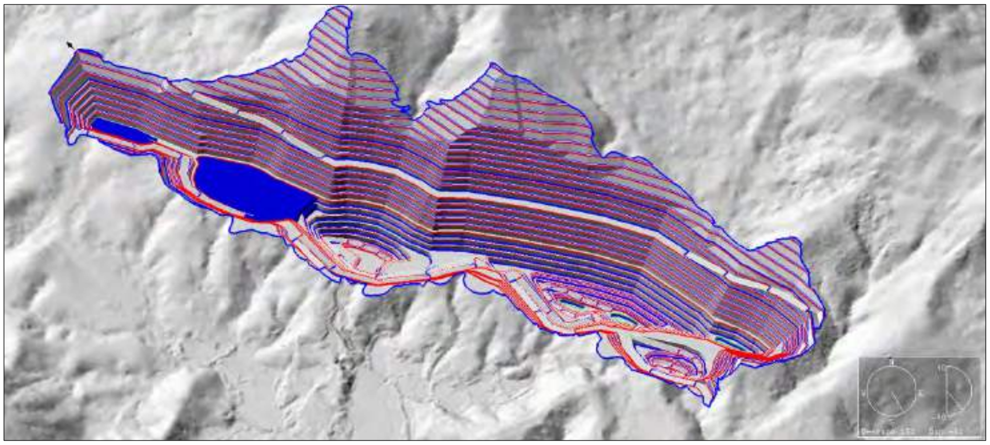
Source: SRK, 2015

Figure 16.9.11: Year 8 Pit Phase Plot



Source: SRK, 2015

Figure 16.9.12: Year 9 Pit Phase Plot



Source: SRK, 2015

Figure 16.9.13: Year 10 Pit Phase Plot

16.10 Overburden Storage

Provision has been made for two overburden storage areas that are defined as the west dump and another as the east dump. Because there are many options available for overburden storage, the west dump is essentially a self-contained and buttressed location next to the TSF location. The location of the west dump can also move to the east rather than expanding northward as is designed in the PEA. The east dump is located where it is so that waste material high up on the pit has a horizontal haul to the dump as opposed to moving the dirt down into the valley. There is the potential that there is mineralized material underneath the dump so sterilization should take place. There is 18.25 Mm^3 of saprolite and hard rock that is required for the construction of the tailings dam facility.

The total storage capacity in the PEA is more than enough for expanded pit operations and given the amount of areas to dump the designs are flexible. The overburden storage capacities of the various facilities are show in Table 16.10.1.

Table 16.10.1: Overburden Storage Areas

Overburden Storage	Volume Mm^3	Assumed Density S.G	Tonnage
East	37.1	2	74.2
West	63.3	2	126.6
Tailings	18.8	2	37.6
Total	**119.2**	**2**	**238.4**
Pit Overburden	111.4	2	222.8

Source: SRK, 2015

16.11 Haulage Profile

A significant design requirement for the production schedule was to mine using a consistent load and haulage fleet that would be able to supply the mill with the required feed, and to not become waste bound. In order to do this, the cycle time and distance must be estimated into the block model so the different haulage lengths from different parts of the pit can be accurately calculated.

Table 16.11.1 details the speed in which the trucks have been estimated to run at different gradients. This is calculated for both loaded and empty portions of the haulage cycle. The speed has been capped at 40 km/h for safety reasons and to not bias the operating speed of the trucks. Loaded truck speeds are capped at 25 km/h.

Table 16.11.1: Rimpull Curve Representing Truck Speeds by Gradient for 777 Truck

Truck 777	Gradient (%)	Speed Uphill (km/h)	Speed Downhill (km/h)
Loaded	Flat	25	25
	0	25	25
	2	18.5	25
	4	17	25
	6	14.9	21
	8	11.5	19
	10	10.4	17
	15	5	5
Empty	Flat	40	40
	0	40	40
	2	31.5	30
	4	27	30
	6	23	24
	8	20	18.5
	10	17.5	15
	15	10	10

Source: SRK, 2015

Haulage is based on material from each phase going to either the east or west waste dump and crusher location. Because there are multiple ramps within the pit, the haulage profile within the pit to an exit point was estimated with an 8% grade, the in-pit time is then added to the time taken defined by haul road strings to the dump or crusher. The cycle time and distance is stored in the block model and the haul cycles are matched with a routing block code in the block model. For example, waste blocks are assigned the waste cycle time based on the distance of the block to the waste dump. Strings to dumps are based on centroid locations.

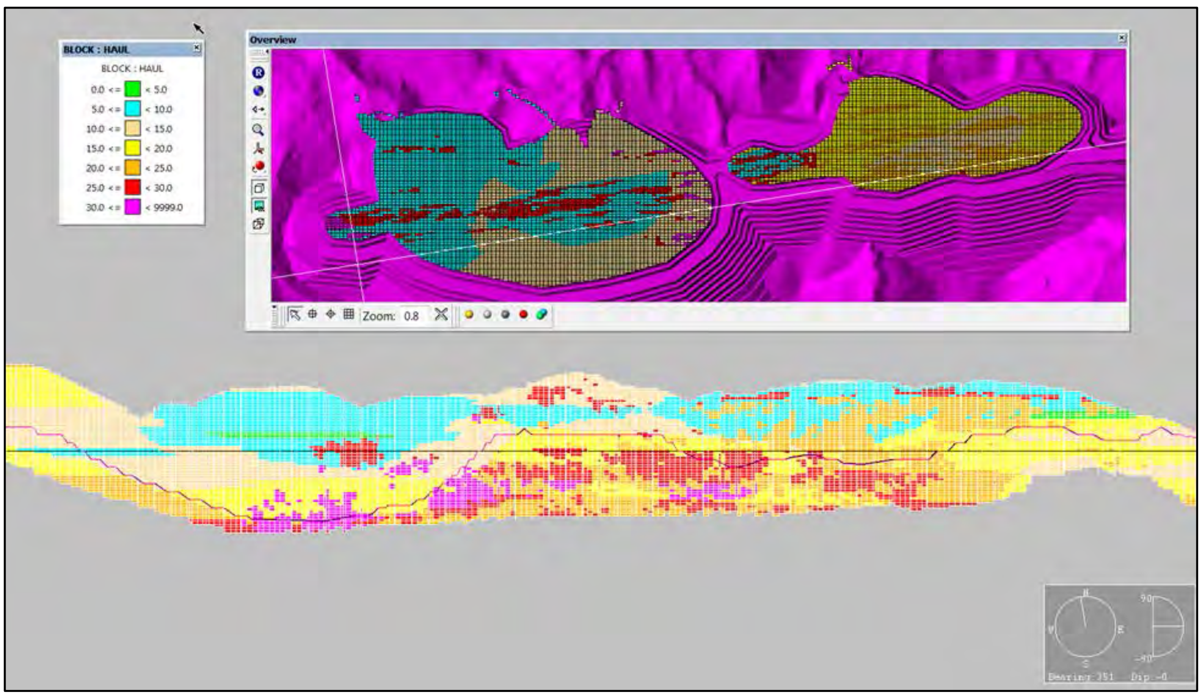
Table 16.11.2 details the cycle times for each mill grade bin and waste broken down by year the mill feed is to be mined. The cycle times shown are maximum efficiency moving cycle times and do not include operational delays that are added to the cycle times for truck spotting, dumping and general delays.

Table 16.11.2: Annual Production and Cycle Time Information that Form Basis of Fleet Estimate

Year	Mill Cycle (m)	Mill Distance (m)	Mill Tonnes	Waste Cycle (m)	Waste Distance (m)	Waste Tonnes	Saprolite Tonnes	Hard Tonnes	% Saprolite
Pre-Strip	30	5,428	450,000	11	2,095	10,550,000	243,293	206,707	54
Year 1	26	4,557	5,500,000	11	1,869	25,000,000	1,627,187	3,872,813	30
Year 2	26	4,437	5,500,000	13	2,195	24,503,648	1,075,826	4,424,174	20
Year 3	23	3,962	5,500,000	13	2,099	24,981,847	563,909	4,936,091	10
Year 4	29	4,958	5,276,367	13	1,845	35,000,000	270,624	5,005,743	5
Year 5	32	5,580	4,500,000	13	1,771	35,000,000	77,171	4,422,829	2
Year 6	35	6,087	4,500,000	14	2,046	31,799,088	146	4,499,854	0
Year 7	27	4,508	6,078,707	16	2,659	26,438,092	246,641	5,832,066	4
Year 8	29	4,713	5,738,690	21	3,361	20,209,005	1,271	5,737,418	0
Year 9	30	4,934	5,138,162	19	2,649	23,922,694	203,076	4,982,936	4
Year 10	22	3,620	5,500,000	17	2,306	20,523,043	10,760	5,489,240	0
Year 11	27	4,394	2,077,569	23	3,251	3,559,789	-	2,077,569	-
Average	**28**	**4,685**	**55,759,494**	**15**	**2,228**	**281,487,205**	**4,319,902**	**51,487,441**	**8**

Source: SRK, 2015

Figures 16.11.1 illustrates the cycles times that have been coded into the block model and associated waste and stockpile locations.



Source: SRK, 2015

Figure 16.11.1: Cross-Section of Haulage Cycle Time

16.12 Fleet Estimate

The fleet estimate generated by SRK is based on an internal spreadsheet that matches equipment with particular product streams. Consumption information for fuel, lube, tires, etc., is based on handbook information for the particular class of equipment and is not company specific.

Because of the large amount of rainfall, hilly terrain, amount of saprolite and expected variability of the mill feed, SRK suggests a mixed fleet be purchased for mill feed and waste mining. The first fleet is comprised of CAT 374 excavators loading articulated dump trucks (ADTs), and will be used for pioneering excavation, saprolite mining and can assist in selective mill feed mining. The articulated mining fleet has lower ground pressure, can work in tight terrain and has six-wheel drive so it is better suited for dealing with slippery saprolite road conditions. As the saprolite is removed the second larger mining fleet of PC2000-8 excavators and CAT 777 trucks will perform the majority of the bulk production. The operators at Montagne d'Or should be mindful that the roads will require a road course and during the feasibility stage of the Project, quarry sources should be found.

Blasthole drilling will be done by top hammer rigs with a blasthole pattern that will support a powder factor of approximately 0.22 kg explosive per tonne of blasted rock. Total LoM capital, including sustaining capital for equipment rebuilds, has been estimated at US$87 million with initial capital estimated at US$54 million for major equipment items.

Table 16.12.1 details the breakdown for total operating hours per year. Each day comprises of two 12 hour shifts.

Table 16.12.1: Operating Hour Assumption for Major Fleet Items

Equipment	Type	Mechanical Availability	Operator Efficiency	Utilization Factor	Operating Hours	Total Operating	Hours Operating Per Year
		%	%	%	hr	%	hr
Drilling	ROC9	85	85	85	8,760	61.4	5,380
Loading	PC2000-8	90	85	85	8,760	65.0	5,696
	CAT374	85	85	85	8,760	61.4	5,380
Hauling	CAT 777	85	85	85	8,760	61.4	5,380
	CAT 740	85	85	85	8,760	61.4	5,380

Source: SRK, 2015

16.12.1 Drilling

The drilling equipment suggested by SRK is only a recommendation given the understanding of the grade distribution, bench height, powder factor and penetration rate. SRK has recommended a moderate diameter blasthole so a 5 m x 5 m blast pattern has a powder factor of 0.22 (kg/t explosive) for a 5 m bench. The details of the drill and blast assumptions are displayed in Table 16.12.1.1.

Table 16.12.1.1: Blasthole Design Criteria

	Units	Waste	Mill Feed
Penetration Rate	m/h	27	27
Hole Diameter	mm	139	139
Hole Area	m^2	0.015	0.015
Bench Height	m	5.00	5.00
Sub-drill Height	m	0.50	0.50
Total Hole Length	m	5.50	5.50
Loaded Length	m	4.40	4.40
Stemming Height	m	1.10	1.10
Loaded Hole Volume	m^3	0.07	0.07
Explosive Loading Density	kg/m	18.21	18.21
Explosive Load Length	m	4.40	4.40
Explosive per Hole	kg	80.12	80.12
Powder Factor	kg/t	0.18	0.22
Tonnage Blasted per Hole	t/hole	408.0	333.8
Volume Blasted per Hole	m^3/hole	138.8	113.6
Square Pattern Spacing	m	5.27	4.77
Drilling Time per Hole	min/hole	12.22	12.22
Non-Productive Time per Op. Hr.	min	5.00	5.00
Tramming and Setup Time per Op. Hr.	min	9.49	9.49
Available Time per Hour	min	45.51	45.51
Holes Drilled per Op. Hour		3.72	3.72
Length Drilled per Op. Hour	m/op.hr	20.48	20.48
Drill Productivity w/out Redrills	t/op.hr	1,519	1,243
Drill Productivity w/ Redrills	t/op.hr	1,443	1,181

Source: SRK, 2015

16.12.2 Load and Haul

The number of load and haul units are dependent on the relationship of each type of loading unit and the cycle time. SRK has suggested that PC2000-8 be used with a bucket capacity of approximately 12 m^3. Mining is anticipated occur on a 5 m flitch or a 15 m bench so the loader is expected to be in excavator configuration rather than shovel configuration.

The primary hauling unit that is attached to the PC2000-8 is expected to be a CAT 777 truck of the 100 t class. The CAT 777 truck has an excellent performance record and well suited to the size of the operation under consideration at Montagne d'Or.

The secondary load and haul fleet for saprolite mining, pioneering and selective mining are CAT 374 excavators working with a CAT 740 ADTs or similar.

Table 16.12.2.1 details the loading design criteria for the two fleets based on rock type. Key parameters include payload weight using volumetric calculations and the number of passes required to load the truck.

Table 16.12.2.1: Loading Design Criteria

	Units	Hard Rock	Saprolite	Hard Rock	Saprolite
Loader		**PC2000-8**	**PC2000-8**	**374K**	**374K**
Truck		**Cat 777**	**Cat 777**	**Cat 740**	**Cat 740**
Heaped Bucket Capacity	m^3	13	13	4.50	4.5
Actual Bucket Capacity	m^3	12.35	12.35	4.41	4.41
Loose Material Density - Wet	wmt/lcm	1.61	2.45	1.61	2.45
Tonnes per Pass	wmt	19.85	30.26	7.09	10.80
Truck Capacity	mt	90.70	90.70	41.00	41.00
Truck Capacity	m^3	60.1	60.1	24.0	24.0
Theoretical Passes - Volume		4.87	4.87	5.44	5.44
Theoretical Passes - Weight		4.57	3.00	5.78	3.79
Actual Passes - Weight		5.00	3.00	6.00	4.00
Truck Load - Volume	m^3	61.8	37.1	26.5	17.6
Truck Load - Weight	wmt	99.2	90.8	42.5	43.2
Truck Load - Weight	t	79.4	86.5	34.0	41.2
Truck Capacity Utilized - Volume	%	102%	61%	110%	73%
Truck Capacity Utilized - Weight	%	109%	100%	103%	105%
Truck Load Time	min	3.80	2.90	4.30	3.20
Trucks Loaded per Hour		15.79	20.69	13.95	18.75
Maximum Productivity - Weight	wmt/hr	1,566	1,878	593	810
Maximum Productivity - Weight	t/hr	1,253	1,788	474	771
Maximum Productivity - Bank	bcm/hr	557	608	210	262
Maximum Productivity - Loose	lcm/hr	780	730	295	315

Source: SRK, 2015

When the production schedule and cycle times from Table 16.11.2 (Annual Production and Cycle Time Information that Form Basis of Fleet Estimate) are matched with the pieces of mine equipment, the LoM fleet estimate is calculated. In Section 16.12.4, Table 16.12.4.1 details the fleet estimate that has been used for capital and operating cost estimates. The numbers in red are predominately low grade stockpile re-handle.

16.12.3 Roads and Dumps

The major support equipment is based on 25% total tonnage for bulldozers operating at 400 loose cubic meter (lcm) per hour, 25% of loader hours for graders and 7% of haul hours for water trucks.

16.12.4 Fleet Estimate

The total fleet estimate does not include ancillary equipment such as tool carriers, light plants or other support equipment.

Table 16.12.4.1: Suggested Mine Equipment Fleet

Period	Type	Drilling	Loading		Hauling		Ancillary		
	Model	ROC 9 (139 mm)	PC2000	CAT 374K	CAT 777 100 t	CAT 740 40 t	CAT D9 Track Dozer	CAT 16 m Grader	Water Truck
-1		1	1	2	3	13	2	1	2
1		3	3	3	14	14	3	2	3
2		3	3	3	16	14	3	2	3
3		4	3	2	15	14	3	2	3
4		5	4	3	21	14	4	2	3
5		5	4	3	21	14	4	2	3
6		5	4	1	21	12	4	2	3
7		4	4	2	21	11	3	2	3
8		4	3	1	20	10	3	2	3
9		4	3	2	21	12	3	1	3
10		4	3	1	16	10	3	2	3
11		1	1	1	4	9	1	1	2
12		0	0	2	0	4	1	1	1

Source: SRK, 2015

16.13 Mine Operating Cost

Mining costs have been estimated using first principles based on the mine production schedule. Because the production schedule includes stockpiles, costs have been split between an in situ mining cost that includes the planned re-handle, and the mining cost that is based on total material movement.

Saprolite has been modeled as requiring drilling for grade control purposes, but waste saprolite is not blasted thus assumed to be freely diggable.

The variable costs associated with particular mine equipment are detailed in Table 16.13.1 and are based on industry surveys for the particular class of equipment.

Table 16.13.1: Equipment Assumptions

Database Description	Unit	ROC9	PC2000	CAT 374	CAT 777	CAT 740	CAT D9	16M	CAT740
Overhaul Labor	man hrs.	0.59	0.55	0.2	0.12	0.08	0.14	0.14	0.18
Maint. Labor	man hrs.	0.49	0.83	0.3	0.23	0.15	0.22	0.26	0.33
Fuel Cons.	L/h	51.41	120.88	58.2	75.93	30.11	73.02	37.22	58.10
Lube Cons.	L/h	3.37	6.71	2.3	3.41	1.68	2.42	1.60	1.88
Tires	US$/h	-	-	-	28.03	11.48	-	2.16	1.59
Overhaul Parts	US$/h	23.63	22.12	7.8	4.34	2.51	5.77	5.56	17.74
Maint. Parts	US$/h	19.33	33.17	11.8	8.07	4.65	8.65	10.32	5.45
Wear Parts	US$/h	11.69	7.73	4.0	-		16.33	1.35	10.11

Source: SRK, 2015

Table 16.13.2 details the mine operating cost by discipline. By dividing the total mine operating cost by the total tonnage the unit operating cost is estimated at US$2.37/t moved, and if the cost is divided by the in situ tonnes with no re-handle material then the cost is US$2.42/t mined.

Table 16.13.2: Mine Operating Cost

Cost	Units	US$
Total Operating Cost	US$000’s	815,076
Total Tonnes In Situ	t	337,306
Total Tonnes Moved (Includes Re-handle)	t	344,490
Operating Cost In Situ Tonnes	US$/t	2.37
Operating Cost In Total Tonnes	US$/t	2.42
Unit Costs (Total Tonnes)		
Drilling	US$/t	0.14
Blasting	US$/t	0.29
Loading	US$/t	0.21
Hauling	US$/t	0.92
Roads & Dumps	US$/t	0.22
Labor	US$/t	0.58

Source: SRK, 2015

Table 16.13.3 details the estimated labor requirements for mine operations and maintenance. These estimates are based on manpower requirements for developed nations such as the US that will be similar to Montagne d’Or, as the Project is essentially part of France and has been costed with the inflated labor rates compared to the neighboring countries. General labor rates were supplied by Nordgold and maintenance requirements are estimate RoM cost guides.

Table 16.13.4 Details the LoM operating labor cost on an annual basis. Numbers in red are predominately mining stockpiles.

Table 16.13.3: Mine Labor Cost

Year	Blasting Men	Equipment Operators Men	Maintenance Men	Mine Staff Men	Total Mining Manpower Men	Mine Staff (US$000's)	Blasting (US$000's)	Equipment Operators (US$000's)	Mine Maintenance (US$000's)	Total Mining Labor Cost (US$000's)
-1	6	126	-	23	155	2,056	234	5,897	-	**8,187**
1	14	232	66	39	351	3,093	543	10,858	2,490	**16,983**
2	14	240	65	39	358	3,093	543	11,232	2,452	**17,320**
3	14	240	66	39	359	3,093	543	11,232	2,490	**17,358**
4	14	299	79	39	431	3,093	543	13,993	2,976	**20,606**
5	14	295	78	39	426	3,093	543	13,806	2,939	**20,381**
6	14	273	74	39	400	3,093	543	12,776	2,789	**19,202**
7	14	262	70	39	385	3,093	543	12,262	2,640	**18,537**
8	14	240	60	39	353	3,093	543	11,232	2,265	**17,133**
9	14	255	65	39	373	3,093	543	11,934	2,452	**18,022**
10	14	217	45	39	315	3,093	543	10,156	1,685	**15,476**
11	14	95	16	39	164	3,093	543	4,446	599	**8,681**
12	0	45	7	15	67	1,234	-	2,106	262	**3,602**
Total						**$37,313**	**$6,206**	**$131,929**	**$26,040**	**$201,487**

Source: SRK, 2015

Table 16.13.4: Annual Operating Cost

		Total	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**
Production Summary															
Waste	kt	**7,185**	0	0	0	0	0	75	63	0	0	0	0	2,485	4,563
Total Waste	kt	**281,498**	10,550	25,000	24,504	24,982	35,000	35,000	31,799	26,438	20,209	23,934	20,523	3,560	0
Total Mill Feed	kt	**55,807**	450	5,500	5,500	5,500	5,276	4,500	4,500	6,079	5,739	5,186	5,500	2,078	0
Total Production	kt	**344,490**	11,000	30,500	30,004	30,482	40,276	39,575	36,362	32,517	25,948	29,120	26,023	8,122	4,563
	in situ mining	**337,306**	11,000	30,500	30,004	30,482	40,276	39,500	36,299	32,517	25,948	29,120	26,023	5,637	0
Total Power, Fuel & Lube Consumption															
Diesel Fuel	kL	**142,036**	4,374	11,528	12,155	11,963	15,324	15,458	14,522	14,133	13,026	13,723	11,203	3,597	1,031
Lube & Oil	kL	**7,036**	226	575	601	596	760	760	722	696	637	672	556	183	51
Total Operating Cost															
Drilling	US$000's	**47,547**	535	3,185	3,311	4,009	6,161	5,183	5,897	5,113	4,305	4,333	4,501	1,014	0
Blasting	US$000's	**100,084**	1,250	7,476	7,250	8,487	12,680	10,759	12,105	10,515	8,884	8,978	9,205	2,495	0
Loading	US$000's	**72,975**	3,225	6,847	6,942	6,697	7,895	8,509	7,002	6,465	5,113	6,001	5,208	1,905	1,165
Hauling	US$000's	**316,320**	10,176	24,171	26,455	26,013	32,874	33,730	31,826	31,685	30,427	31,909	25,406	9,144	2,503
Roads & Dumps	US$000's	**76,662**	3,654	6,202	6,443	6,572	8,213	8,113	7,487	7,071	6,189	6,753	5,875	2,398	1,690
Labor (from labor sheet) Incls. G&A	US$000's	**201,487**	8,187	16,983	17,320	17,358	20,606	20,381	19,202	18,537	17,133	18,022	15,476	8,681	3,602
Operating Cost	US$000's	**815,076**	27,027	64,865	67,723	69,135	88,429	86,675	83,519	79,386	72,051	75,997	65,672	25,637	8,960
Operating Cost	US$/t	**$2.37**	$2.46	$2.13	$2.26	$2.27	$2.20	$2.19	$2.30	$2.44	$2.78	$2.61	$2.52	$3.16	$1.96
Unit Costs															
Drilling	US$/t	**$0.14**	$0.05	$0.10	$0.11	$0.13	$0.15	$0.13	$0.16	$0.16	$0.17	$0.15	$0.17	$0.12	0
Blasting	US$/t	**$0.29**	$0.11	$0.25	$0.24	$0.28	$0.31	$0.27	$0.33	$0.32	$0.34	$0.31	$0.35	$0.31	$0.00
Loading	US$/t	**$0.21**	$0.29	$0.22	$0.23	$0.22	$0.20	$0.22	$0.19	$0.20	$0.20	$0.21	$0.20	$0.23	$0.26
Hauling	US$/t	**$0.92**	$0.93	$0.79	$0.88	$0.85	$0.82	$0.85	$0.88	$0.97	$1.17	$1.10	$0.98	$1.13	$0.55
Roads & Dumps	US$/t	**$0.22**	$0.33	$0.20	$0.21	$0.22	$0.20	$0.21	$0.21	$0.22	$0.24	$0.23	$0.23	$0.30	$0.37
Labor (from labor sheet) Incls. G&A	US$/t	**$0.58**	$0.74	$0.56	$0.58	$0.57	$0.51	$0.51	$0.53	$0.57	$0.66	$0.62	$0.59	$1.07	$0.79

Source: SRK, 2015

16.14 Mine Capital Cost

Mining equipment capital cost was sourced from cost guides that are traditionally higher than that achieved by mining companies who have the capability and volume to reduce unit prices through vendor negotiation. That being said, the prices have enough contingency to cover transportation, duty and erection etc. that have not been added to list prices at this level of study. To determine the sustaining capital in the form of rebuilds and equipment replacement, Table 16.14.1 details the rebuild cost assumption, hours needed before the rebuild and hours before the equipment is replaced.

Table 16.14.1: Mine Capital and Sustaining Capital Conditions

Make/Model		ROC D9	PC2000-8	Cat 374	Cat 777	Cat 740	Cat D9	16M	Water Truck
Size		139	12	5	90	40	400	5	37,000
Units		mm	m^3	m^3	wmt	wmt	lcm/h	m	L
Final Unit Price	US$000's	888	3,428	1,079	1,368	710	832	832	710
Rebuild Capital Percent	%	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Rebuild Hours	hrs.	15,000	30,000	30,000	30,000	30,000	15,000	15,000	45,000
Replace Hours	hrs.	60,000	80,000	80,000	80,000	75,000	45,000	45,000	75,000

Source: SRK, 2015

Table 16.14.2 details the initial capital of US$54 million by major equipment type. The sustaining capital has been estimated at US$33 million for new purchases and major rebuilds based on accumulated equipment hours through the LoM.

Table 16.14.2: Initial and Sustaining Capital Estimate

Year	Drilling (US$000's)	Loading (US$000's)	Hauling (US$000's)	Roads & Dumps (US$000's)	Capital Cost (US$000's)
-2	888	5,586	13,334	5,411	25,219
-1	1,776	7,935	15,758	2,825	28,294
1	-	-	2,736	-	2,736
2	888	-	-	70	958
3	888	3,428	6,840	1,584	12,740
4	888	-	-	1,818	2,706
5	-	2,742	7,450	-	10,192
6	-	-	-	426	426
7	-	-	-	664	664
8	-	216	-	399	614
9	710	-	-	550	1,261
10	-	-	-	-	-
11	-	-	-	845	845
Total	**$6,038**	**$19,907**	**$46,118**	**$14,593**	**$86,656**

Source: SRK, 2015

17 Recovery Methods

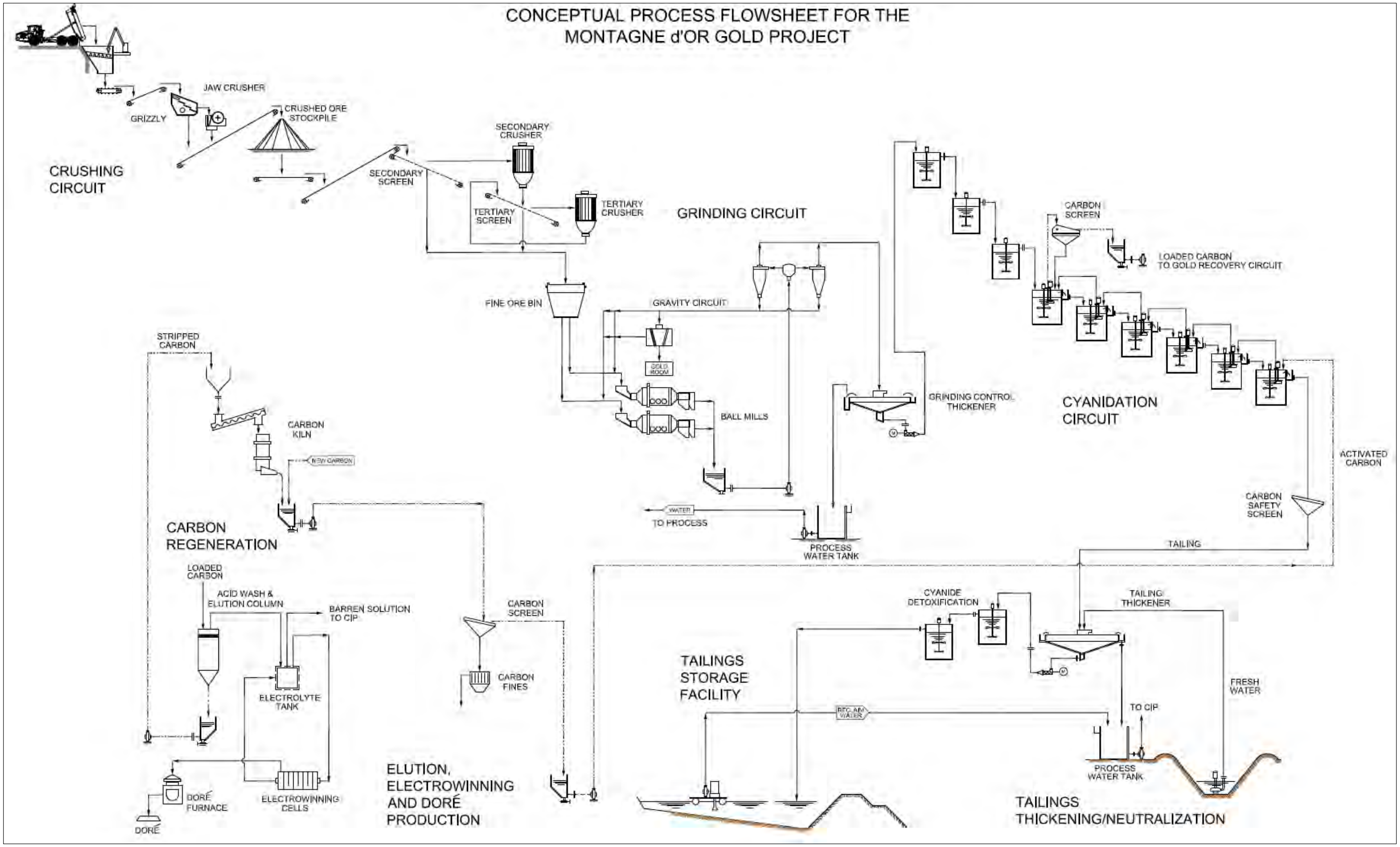
Metallurgical testwork was conducted to evaluate three different process flowsheet options including:

- Whole-ore cyanidation;
- Gravity concentration followed by cyanidation of the gravity tailing; and
- Gravity concentration followed by gold and silver flotation from the gravity tailing and cyanidation of the flotation concentrate.

After conducting a trade-off study the process flowsheet that includes gravity concentration followed by cyanidation of the gravity tailing was selected as this flowsheet option offers higher overall gold and silver recoveries and resulted in the highest Project NPV and highest IRR. Trade-off study results presented in Section 13.10.

17.1 Process Description

The selected process flowsheet will include gravity concentration followed by cyanidation of the gravity tailings to recover the contained gold and silver values, and will incorporate process unit operations that are standard to the industry, including: crushing, grinding, agitated cyanide leaching, gold and silver adsorption onto activated carbon, gold and silver desorption, electrowinning and refining. A conceptual process flowsheet is shown in Figure 17.1.1. Preliminary process design criteria are presented in Table 17.1.1 and a major equipment list is provided in Table 17.1.2.



C:\Users\hvialpando\Documents_SRK Denver\SRK Staff\Eric Olin\176302.Vol.I.Fresh.FS.Simplified.Flow.Plaba.Process.Plant.dwg

Source: SRK, 2015

Figure 17.1.1: Conceptual Process Flowsheet

Table 17.1.1: Preliminary Design Criteria for the Montagne d'Or Process Plant

Area	Units	Criteria
Mine Production		
Mine Production Rate, t/y	t/y	4,500,000
Operating days per year	days	365
Mill Design Capacity	t/d	12,500
Gold Grade	g/t	1.8
Crushing		
Operating days per year	days	365
Shifts per day		2
Hours per shift	hours	12
Availability	%	70
Operating Hours per day	hours	17
Crushing Rate	t/h	744
Feed F80	mm	1,200
Product P80	mm	9.5
Grinding		
Operating days per year	days	365
Hours per day	hours	24
Shifts per day		2
Hours per shift	hours	12
Availability	%	92
Feed Rate	t/h	566
Ball Mill Work Index, Bond, BWi	kWh/t	12.0
SMC, Axb		33.0
Abrasion Index , Ai		0.1
Ball Mill, F80	µm	9,500
Ball Mill, P80	µm	74
Grinding Control Thickener		
Slurry Feed Density	w/w%	35
Underflow density	w/w%	60
Specific Settling Area	$m^2/t/d$	0.20
Hydraulic Loading	$m^3/m^2.hr$	4.50
Flocculant (Hychem AF 303)	g/t	20
Cyanidation		
Slurry Density	w/w%	45
Retention Time	%	48
Cyanide Leach Concentration (NaCN)	ppm	500
PH		10.5
Cyanidation Tailing Thickening		
Slurry Feed Density	w/w%	45
Underflow density	w/w%	55
Specific Settling Area	$m^2/t/d$	0.20
Hydraulic Loading	$m^3/m^2.hr$	4.50
Flocculant (Hychem AF 303)	g/t	20
Cyanide Detoxification		
Slurry density	w/w%	55
Retention time	hours	2
Feed CN_T	ppm	TBD
Discharge CN_T	ppm	<1
SO_2 dosage	g/g CNT	TBD
$Na_2S_2O_5$ (SO_2 Equivalent)	g/g CNT	TBD
$CuSO_4$	g/g CNT	TBD

Source: SRK, 2015

Table 17.1.2: Process Plant Major Equipment List

Description	Quantity	Units	Size	KW	Comment
Crushing Circuit					
RoM Mineralized Material Bin	1	t	250		
Stationary Grizzly	1	mm	600 x 600		Spacing
Apron Feeder	1	mm	1,220 x TBD	20	
Fines Vibrating Screen	1	m	2.4 x TBD	50	
Primary Jaw Crusher	1	mm	1,220 x 1,600	150	
Crushed Ore Stockpile	1	t	6,000		One shift live capacity
Secondary Cone Crusher	1		MP1000	750	Standard
Secondary Vibrating Screen	2	m	2.4 x TBD	50	50mm bottom deck
Tertiary Cone Crusher	2		MP1000	750	Short head
Tertiary Vibrating Screen	4	m	2.4 x TBD	50	9.5 mm bottom deck
Grinding Circuit					
Fine Ore Bin	1	t	6,000		
Ball Mill	2	m	4.5 x 9	5,000	
Cyclones	8+4	inch	D-15		
Centrifgual Gravity Concentrator	2		KC-XD48	75	Knelson or equivalent
Grinding Control Thickener	1	m	50	7.5	Conventional
Cyanidation Circuit					
Agitated Leach Tanks	9	m	18 x18	200	
Interstage Screen	6		TBD		Vertical air swept
Carbon Recovery Screen	1		TBD		
Carbon Safety Screen	1	m	3 x 8	2 x 15	
Tailings Thickener	1	m	50	7.5	Conventional
Detox Circuit					
Agitated Detoxifcation Tanks	2	m	10 x10	50	
Gold Room					
Acid Wash Column	2	M	1.2 x 9.5		4.5t carbon capacity
Elution Column	2	m	1.2 x 9.5		4.5t carbon capacity
Carbon Activation Kiln	1		250 kg /hr		Horizontal, diesel fired
Electrowinning Cells	6	amphere	1,500		SS 304/polylined
Dore' Furnace	1				Diesel Fired

Source: SRK, 2015

17.1.1 Run-of-Mine Pad

Run-of-mine (RoM) material will be direct truck dumped into the primary crusher to the extent possible. A front-end loader will be used to reclaim excess material from the various stockpiles to the RoM hopper feeding the primary crusher.

17.1.2 Crushing Circuit

RoM material will be loaded into the RoM feed hopper by haul truck or front-end loader. A grizzly will be fitted to the RoM hopper to protect the downstream equipment from oversize material. A rock breaker will be provided to reduce oversize rock such that it will pass through the grizzly. RoM material will be drawn from the hopper at a controlled rate by a variable speed apron feeder and discharge onto a vibrating grizzly ahead of the jaw crusher. The grizzly oversize (+6 inches) will discharge to the jaw crusher.

The crusher product and vibrating grizzly undersize will discharge onto a conveyor belt and be transported to the crushed material stockpile. Crushed material will be withdrawn from the stockpile at a controlled rate by variable speed apron feeders and conveyed to the secondary and tertiary crushing circuit where the ore will be crushed to 80% passing (P_{80}) 9.5 mm. The crushed material will be conveyed to a fine ore bin ahead of the grinding circuit.

17.1.3 Grinding, Classification and Gravity Circuit

The grinding circuit will consist of two ball mills operated in closed circuit with hydrocyclones to produce a final grind size of P_{80} 75 µm. The cyclone underflow will flow to either the Knelson-type semi-continuous centrifugal gravity concentrators, or be recirculated back to the ball mills. The cyclone overflow will gravitate to the pre-leach thickener where it will be thickened prior to being pumped to the cyanidation circuit. Coarse gold will be recovered by the Knelson gravity concentrators and further upgraded in the gold room with a shaking table to produce a concentrate suitable for smelting. The tails from the Knelson concentrators will flow back to the ball mill. It should be noted that a ball mill grinding circuit has been selected rather than a SAG circuit, due to the high resistance to SAG mill grinding indicated by the SMC tests performed on the two master composites.

17.1.4 Grinding Control Thickener

Cyclone overflow will gravitate to the trash removal screen. The trash screen will remove any coarse particles, wood fragments, organic material, plastics and lime slurry grits that could otherwise blind the inter-tank screens. The screen oversize (trash) will be collected in a bunker or bin, and the screen undersize (slurry) will gravitate to the grinding control thickener where it will be combined with flocculant in the feed well. Flocculant fed to the thickener will be diluted with water in a static mixer to ensure good dispersion throughout the feed stream. Thickener underflow will be pumped to the CIP circuit, and thickener overflow will report to the process water tank.

17.1.5 Leach and Carbon Adsorption Circuit

The thickener underflow will be pumped to the leach distributor feed box passing through a two stage cross cut feed sampler along the way. The sampler will be used to take representative samples of the feed head grade for metallurgical accounting purposes. It is anticipated that the cyanidation circuit will consist of three agitated leach tanks and six CIL tanks. The tanks will be interconnected with launders, and slurry will flow by gravity through the tank train. Each tank will be fitted with a dual impeller mechanical agitator to ensure uniform mixing and dispersion. Oxygen required for leaching will be provided by air sparging through the bottom of the agitator shaft into the slurry. The adsorption tanks will each be fitted with an air swept woven wire inter-tank screen to retain the carbon. All tanks will be fitted with bypass facilities to allow any tank to be removed from service for agitator or screen maintenance.

Sodium cyanide solution will be metered into the leach feed distribution box, as required, to maintain the desired cyanide concentration (500 ppm NaCN) in the circuit. Compressed air will be distributed to the circuit and sparged down the shafts of the agitators to allow a high dissolved oxygen profile to be maintained in the circuit. Fresh and regenerated carbon will be returned to the circuit at CIL Tank 6, and will be advanced counter-current to the slurry flow by pumping slurry and carbon from Tank 6 to Tank 5 to Tank 4, and so on. The inter-tank screen in each CIL tank will retain the carbon and allow the slurry to gravity flow to the next CIL tank. This counter-current process will be repeated until the carbon eventually reaches CIL Tank 1 at which point an air lift will be used to transfer loaded carbon to the loaded carbon recovery screen. The loaded carbon will be washed and dewatered on the recovery screen prior to reporting to the acid wash/elution column. The recovery screen undersize will return to the CIL circuit.

Slurry from the last CIL tank (leach tails) will gravitate to the vibrating carbon safety via the tails sampler for metallurgical accounting. The safety screen will recover any carbon leaking through worn inter-tank screens or overflowing the tanks. Screen underflow will gravitate to the cyanide destruction circuit via the cyanide destruction distribution box. Barren carbon returning to the adsorption circuit from the carbon regeneration kiln will be screened on the sizing screen to remove fine carbon and prevent associated gold losses. The sized and regenerated carbon will report to CIL Tank 6, or alternately to Tank 5. The CIL tanks will be located in a bunded area with a sloping concrete floor. Any spillage from the circuit will report to one of two sumps and can be returned to the circuit or to the carbon safety screen ahead of the cyanide destruction circuit.

17.1.6 Elution and Gold Room Operations

The following operations will be carried out in the elution and gold room areas:

- Acid washing of carbon;
- Stripping of gold from loaded carbon;
- Electrowinning of gold from pregnant solution;
- Smelting; and
- Carbon regeneration.

Acid Wash

Loaded carbon will be recovered on the loaded carbon recovery screen and directed to the acid wash column. Acid washing of the carbon will commence after carbon transfer and drain down is complete. The acid wash solution, 3% w/w HCl in fresh water, will be mixed in the dilute acid tank and transferred to the acid wash column. The acid wash process removes contaminants, primarily calcium, from the loaded carbon and prevents carbon fouling which reduces the effectiveness of the carbon. After the prescribed acid soak period, the carbon will be rinsed with fresh water. Approximately three bed volumes of fresh water will be pumped through the column to displace any residual acid from the carbon. Dilute acid and rinse water will be neutralized and disposed of with the tailings. Acid-washed carbon will be transferred to the elution column for stripping.

Pre-Soak and Elution

Strip solution will be pumped from the stripping water tank through inline heater exchangers into the base of the elution column. Sodium hydroxide and sodium cyanide solutions will be pumped from the respective storage tanks into the stripping water tank. The loaded carbon will be pre-soaked in the 2% cyanide / 2% caustic solution for 30 minutes to prepare the gold for elution. The carbon will then be eluted by hot strip solution (120^0 C), which will pass out of the circuit to the pregnant solution tank. Outgoing strip solution will pass through the recovery heat exchanger to heat the incoming strip solution.

Electrowinning

Direct current will be passed through stainless steel anodes and stainless steel wool mesh cathodes to deposit gold and silver sludge on the cathodes. The electrowinning cells, arranged in parallel, will contain 12 cathodes each to provide a high cell pass efficiency to ensure a minimum gold tenor in the barren eluate. Solution discharging from the electrowinning cells will return by gravity to the pregnant solution tank. Electrowinning will continue until the solution exiting the electrowinning cells is depleted of gold.

Gold Room

The electrowinning cells will be located within the security area of the gold room. Rectifiers, one per cell, will be located in a non-secure area below the cells allowing maintenance access without breaching gold room security. The electrowon gold and silver will be removed from the cathodes in situ by washing with high pressure water. The resulting sludge will be filtered in laboratory style pressure filters and dried in an oven. The sludge will then be direct smelted with fluxes in an HFO or diesel fired furnace to produce doré bars. Slag from smelting operations will be returned to the milling circuit. Fume extraction equipment will be provided to remove gases from the cells, oven and smelting.

Carbon Regeneration

After completion of the elution process, the barren carbon will be transferred from the elution column to the carbon dewatering screen to dewater the carbon prior to entering the feed hopper of the horizontal carbon regeneration kiln. Any residual water will be drained from the carbon in the kiln feed hopper before it enters the kiln. It is anticipated that only 75% of the carbon will be regenerated each cycle. In the kiln, the carbon will be heated to 650°C to 750°C for 20 minutes to allow regeneration to occur. Regenerated carbon from the kiln will be quenched and report to the carbon sizing screen. The screen oversize (regenerated and sized carbon) will return to the cyanidation circuit.

17.1.7 Carbon Safety Screen

Tailings slurry from the final CIL tank will gravitate through the metallurgical sampler to the carbon safety screen. Recovered carbon will be collected in the fine carbon bin for potential return to the circuit. A two stage cross cut feed sampler will be used to take representative samples of the tails for metallurgical accounting purposes. The safety screen undersize (leached slurry) will be forwarded to the cyanide destruction circuit.

17.1.8 Cyanide Destruction Circuit

The carbon safety screen undersize slurry will report to the SO_2 / air cyanide destruction circuit. The slurry will flow from the cyanide destruction distribution box to the first cyanide destruction tank. The cyanide destruction circuit will reduce the weak acid dissociable cyanide (CNwad) concentration in the CIL discharge to less than 1 ppm. The cyanide destruction circuit will consist of two agitated tanks each with one hour residence time.

The detoxification process utilizes SO_2 and air in the presence of a soluble copper catalyst to oxidize cyanide to the less toxic compound cyanate (OCN). The SO_2 source will be SMBS. Copper sulfate pentahydrate will be added to supply the necessary copper in solution. Air will be sparged into the cyanide destruction tanks through the agitator shaft. Slaked lime will be added to neutralize the sulfuric acid formed in the reaction and maintain a level of approximately 9 pH.

17.1.9 Tailings Disposal

Tailings from the cyanide detoxification circuit and other miscellaneous waste streams from the process plant will be combined in the tailings collection sump and pumped to the TSF for disposal.

17.2 Consumable Requirements

Lime

Quicklime will be delivered to the site in bulk by pneumatic tanker and stored in the lime silo. It is anticipated that the quicklime will be slaked in a vendor supplied package accompanying the silo. The slaked lime will be pumped to the grinding circuit and the cyanide destruction circuit in a ring main. A dust collector will minimize dust emissions during silo filling.

Cyanide

Sodium cyanide will be delivered as briquettes in shipping containers containing approximately 1 t of cyanide each. The containers will be emptied into the cyanide mixing tank and combined with water to dissolve the cyanide to a target strength of 20% NaCN. Sodium hydroxide will be added to the mixing tank prior to cyanide addition in order to maintain a solution pH of 11 to prevent HCN generation. The mixed cyanide solution will be transferred to the storage tank for dosing to the process. Empty cyanide containers will be returned to the vendor.

Caustic

Caustic (sodium hydroxide) will be delivered to site in bulk bags of pellets. Caustic bulk bags will be lifted by forklift to a small platform at the mixing level. Bags will be emptied by a beak breaker into the mixing tank via a rotary vane feeder to prevent splash back from the tank. Caustic will be transferred to the storage tank for dosing to the process.

Hydrochloric Acid

Concentrated hydrochloric acid (32% w/w) will be delivered to site in 1,000 L isotainers. The concentrated hydrochloric acid will be transferred from the isotainer to the dilute acid mixing and storage tank by a peristaltic pump. Fresh water will be added to dilute the acid to 3% prior to transfer to the acid wash column. This batch process will repeat for each carbon wash cycle.

Activated Carbon

Activated carbon will be delivered in 500 kg bulk bags. Carbon will be added to the carbon quench vessel as required for carbon make-up to the CIL inventory. This addition point will allow removal of carbon fines prior to entering the CIL tanks.

Grinding Media

Grinding balls will be delivered to site in bulk or 200 L steel drums. The balls will be charged to the SAG mill via the SAG mill feed conveyor using a front end loader.

Flocculant

Flocculant for use in the grinding control and tailings thickeners will be delivered to site in 25 kg bags. Flocculant will be added to the flocculant plant storage hopper manually. The vendor supplied flocculant mixing plant will automatically mix batches of flocculant and transfer the mixed flocculant to the aging tank after each mixing cycle is complete. Flocculant will be distributed to the thickeners using positive displacement dosing pumps.

Copper Sulfate

Copper sulfate will be delivered in 1 t bulk bags and will be added to the mixing tank using an electric hoist and bag breaker. Fresh water will be added to the mixing tank to dilute the copper sulfate. The solution will be metered to the cyanide destruction and flotation circuits directly from the mixing tank.

Sodium Metabisulfite

Sodium metabisulfite will be delivered in 1 t bulk bags and will be added to the mixing tank using an electric hoist and bag breaker. An air exhaust fan will draw dust and fumes away from this area as SO_2 gas is evolved and the dust can cause skin irritation. Fresh water will be used to mix the sodium metabisulfite. The solution will be pumped from the mixing tank to the storage tank for metering to the cyanide destruction circuit by dosing pump.

17.3 Metallurgical Accounting

A weightometer on the primary crusher discharge conveyor will measure the primary crushed tonnage. A weightometer on the ball mill feed conveyors will determine mill feed tonnes.

Density and flow meters on the leach feed will allow the dry tonnage of solids to be determined as a cross check on the mill feed tonnage determined from the mill feed weightometer. In conjunction with the leach feed and tails samplers, the mass flow measurements will allow the gold recovered in the CIP to be calculated.

Routine sampling of the leach feed stream and the final leach tailings will ensure reliable composite shift samples for leach head grade and tails solution and residue grades. Regular in-circuit gold surveys will allow reconciliation of precious metals in feed compared to doré production.

17.4 Operating Cost Estimate

Process operating costs are summarized in Table 17.4.1 and are estimated at US$14.55/t processed. Operating costs have been estimated by major category (labor, power, consumables, etc.) and are based on a throughput capacity of 12,500 t/d. The major contributors to operating cost are power and reagents.

Table 17.4.1: Summary of Process Plant Operating Costs

Area	US$/t
Labor	1.50
Comminution Consumables	1.70
Reagents	4.50
Power	6.00
Maintenance Supplies	0.50
Other	0.35
Total Processing Costs	**$14.55**

Source: SRK, 2015

17.4.1 Labor

Labor costs are estimated US$1.50/t and are based on the manpower schedule shown in Table 17.4.1.1. A total of 89 process plant employees (operations and maintenance) have been identified. The labor cost estimate is based on the process plant operating two 12-hour shifts per day and includes a 35% burden rate. Labor rates are based on wages currently paid at mines operated in the southwest USA.

Table 17.4.1.1: Process Plant Manpower Schedule and Estimated Labor Cost

Position	Number	US$/hour	Hours/Yr	Annual, US$	Burden	Annual, US$ (Burdened)	Total, US$
Mill Manager	1			210,000	0.35	283,500	283,500
Administrative Assistant	1	18	2,080	37,440	0.35	50,544	50,544
Subtotal (Mill Manager)	**2**			**247,440**		**334,044**	**334,044**
Operating Crews							
Shift Supervisors	4			80,000	0.35	108,000	432,000
Crusher/Conveying Area Lead Operator	4	25	2,080	52,000	0.35	70,200	280,800
Assistant Crusher/Conveying Area Operator	4	18	2,080	37,440	0.35	50,544	202,176
Grinding/Gravity Area Lead Operator	4	28	2,080	58,240	0.35	78,624	314,496
Leach Circuit Lead Operator	4	28	2,080	58,240	0.35	78,624	314,496
Gold Room Supervisor	2	30	2,080	62,400	0.35	84,240	168,480
Leach / Gold Room Operator	4	25	2,080	52,000	0.35	70,200	280,800
Reagent Area Operator	2	18	2,080	37,440	0.35	50,544	101,088
General Laborer	8	16	2,080	33,280	0.35	44,928	359,424
Control Room Operator	4	28	2,080	58,240	0.35	78,624	314,496
Subtotal (Operating Crews)	**40**			**$529,280**		**714,528**	**$2,768,256**
Senior Metallurgist	1			120,000	0.35	162,000	162,000
Junior Metallurgist	1			80,000	0.35	108,000	108,000
Metallurgical Technician	2	30	2,080	62,400	0.35	84,240	168,480
Chemist	2	28	2,080	58,240	0.35	78,624	157,248
Sample Preparers	6	18	2,080	37,440	0.35	50,544	303,264
Analytical Technicians	6	18	2,080	37,440	0.35	50,544	303,264
Subtotal (Other)	**18**			**$395,520**		**$533,952**	**1,202,256**
Process Plant - Total	**60**			**$1,172,240**		**$1,582,524**	**$4,304,556**
Maintenance - Process Plant							
Maintenance Manager	1			200,000	0.35	270,000	270,000
Maintenance Foreman	4			80,000	0.35	108,000	432,000
Maintenance Planner / Foreman	2	30	2,080	62,400	0.35	84,240	168,480
Mechanics	8	25	2,080	52,000	0.35	70,200	561,600
Welders	2	25	2,080	52,000	0.35	70,200	140,400
Electrician	6	28	2,080	58,240	0.35	78,624	471,744
Instrument Technician	2	28	2,080	58,240	0.35	78,624	157,248
Trades Assistants	4	16	2,080	33,280	0.35	44,928	179,712
Subtotal (Maintenance - Process Plant)	**29**			**$596,160**		**$804,816**	**$2,381,184**
TOTAL Processing Plant + Maintenance	**89**			**$1,768,400**		**$2,387,340**	**$6,685,740**
Feed Tonnes Per Year	4,500,000					**US$/t**	**1.50**

Source: SRK, 2015

17.4.2 Consumables

Comminution consumables are estimated at US$1.70/t and are based on wear liner and grinding media consumption rates that are typical for this type of process facility. As shown in Table 17.4.2.1, reagent costs are estimated at US$4.50/t and are based on reagent consumption rates established during metallurgical testing. Cyanide is the highest cost reagent at US$2.76/t processed and is based on a unit consumption rate of 0.85 kg/t. Cyanide detoxification reagent costs are based on detoxification studies conducted on flotation concentrate residues and will need to be confirmed during the next phase of study. Other costs are estimated at US$0.40/t and are intended to cover costs for miscellaneous consumables, such as carbon, sodium hydroxide, hydrochloric acid and fluxing agents.

Table 17.4.2.1: Estimated Reagent Consumption and Cost

Reagent	Usage (kg/t)	Cost (ex works) (US$/kg)	Freight (US$/kg)	Cost @ Site (US$/kg)	Unit Cost (US$/t)
Cyanide	0.85	2.75	0.50	3.25	2.76
Lime	0.20	0.12	0.50	0.62	0.12
Flocculant	0.02	4.70	0.50	5.20	0.10
CN Destruct - Sodium metabisufite	0.65	0.90	0.50	1.40	0.91
CN Destruct - Copper sulfate	0.060	3.00	0.50	3.50	0.20
Other					0.40
Total Reagent Costs					**4.50**

Source: SRK, 2015

17.4.3 Power

The process plant power cost is estimated at US$6.00/t and is based on on-site power generation at a cost of US$0.20/kWh and a total unit power consumption of 30 kWh/t, which is typical for process plants of this type. The unit power generation cost is based on fuel oil consumption at 0.25 L/kWh and a delivered fuel oil price of US$0.80/L. The estimated unit power consumption includes an estimate of 14 kWh/t to grind feed material to the target grind of P_{80} 75 µm prior to cyanidation, which is based on BWi determinations conducted during the metallurgical investigation.

17.4.4 Maintenance Supplies

Maintenance supply costs are estimated at US$0.35/t and are based on 3% of estimated equipment capital expenditure.

17.5 Capital Cost Estimate

The capital cost for the 12,500 t/d process plant is summarized in Table 17.5.1 and is estimated at US$136.7 million and is considered at a conceptual level with a +/-50% level of accuracy. The capital cost estimate is based on Infomine’s CostMine Model for a CIP processing plant, and includes the following adjustments:

- Capital cost has been escalated to the 12,500 t/d design using the industry accepted Cost-Capacity relationship;
  - o $Cost_{p2} = Cost_{p1} X (Capacity_{p2}/Capacity_{p1})^{0.65}$;
- TSF capital cost has been excluded (treated as a separate cost area);

- Working capital has been excluded (included in the technical economic model); and
- Process plant capital cost has been increased by 30% based on SRK's experience with the CostMine models.

Table 17.5.1: Process Plant Capital Cost Estimate (US$000s)

By Category	**US$000's**
Equipment	48,269
Installation Labor	30,630
Concrete	3,965
Piping	12,717
Structural Steel	4,376
Instrumentation	3,008
Insulation	1,504
Electrical	6,153
Coatings & Sealants	547
Mill Building	8,204
Engineering/Management	17,366
Total (by Category)	**$136,741**
By Area	**US$000's**
Comminution	42,350
CIP Leaching	30,240
Solid-Liquid Separation	8,986
General	10,480
Engineering/Management	13,129
Total (by Area)	**$105,185**
Capital Cost Adjustment (30%)	30%
Total Process Capital Cost	**$136,741**

Source: SRK, 2015

- Working Capital Excluded;
- TSF Starter Dam Excluded;
- CIP capacity escalation factor = .65; and
- Info Mine Model Capital Cost Adjustment Factor = 30%.

17.6 Significant Factors

- The selected process flowsheet will include gravity concentration followed by cyanidation of the gravity tailings to recover the contained gold and silver values, and will incorporate process unit operations that are standard to the industry, including: crushing, grinding, agitated cyanide leaching, gold and silver adsorption onto activated carbon, gold and silver desorption, electrowinning and refining.
- Process operating costs are estimated at US$14.55/t processed. Operating costs have been estimated by major category (labor, power, consumables, etc.) and are based on a throughput capacity of 12,500 t/d. The major contributors to operating cost are power and reagents.
- The capital cost for the 12,500 t/d process plant is estimated at US$136.7 million and is considered at a conceptual level with a +/-50% level of accuracy.

18 Project Infrastructure

Montagne d'Or study is at a preliminary economic level, therefore little in the way of detailed infrastructure analysis has been carried out as no detailed engineering on potential process plant drawings has been conducted. As such, the following section provides some suggestions on what will be required for consideration in further studies.

18.1 Infrastructure and Logistic Requirements

18.1.1 On-Site Infrastructure

The process plant location has been preliminary selected as shown in Figure 18.2.2. Nordgold intends to conduct geotechnical investigation of the proposed site in 2015 to confirm the selection..

The on-site infrastructure design should include:

- Overall site orientation and layout;
- On-site service roads and creek crossings;
- Water supply and treatment;
- Power supply and distribution;
- Mine support facilities;
- Process support facilities;
- Entry station; and
- The man camp facilities.

Criteria for selection of these locations will be to provide sufficient space and size for the process and mine facility pads in close proximity to the mine pit and TSF facility while maintaining a safe elevation above the water surface elevation from the 100-year flood event. Site access roads which interconnect the various site services and areas should be segregated to the maximum extent possible from the mine haul roads.

18.1.2 Site Water Management

There are two distinct wet and two distinct dry seasons each year. Rainfall is significant, averaging over 2 m/y. It is a net positive hydrologic environment with precipitation exceeding evaporation of 0.855 m/y. Therefore, the water management strategy for the Project will be to collect mine impacted water, and use best management practices like sediment collection ponds to both attenuate the flood event peak and to remove sediment from mine impacted water before it is released to the environment. Water will be recycled to the maximum extent possible from the tailings storage facility back to the process plant. When discharge is required, SRK allocated US$15 million for reverse osmosis or other treatment technologies. The final determination of the water treatment will require detailed geochemical characterization and site water balance to size the facility correctly.

18.1.3 Service Roads and Bridges

Montagne d'Or is located in the north-western portion of French Guiana, not far from the Maroni River that forms the border with Surinam. The property is accessible throughout the year by charter aircraft and by road that requires maintenance and upgrade. At Camp Citron, where the base camp is located at a distance of approximately 4 km from the Prospect area, there is a 500 m grass runway that can accommodate small aircraft. Alternatively, a helicopter charter service is available from Cayenne.

The flight from Cayenne to Paul Isnard takes approximately 55 minutes.

A forest road leads for a distance of approximately 125 km from Saint-Laurent-du-Maroni on the Maroni River to the Montagne d'Or prospect area. The first 65 km from Saint-Laurent-du-Maroni to Croisée d'Apatou is maintained by the state and supports all season travel. SOTRAPMAG has an exclusive right to use of the final 60 km of the road, which is currently being maintained by previous project owners to accommodate normal vehicle access for servicing the site.

SOTRAPMAG commissioned a road study in 2014 to assess the Croisée d'Apatou to Citron stretch of the road (60 km) and evaluate the costs of upgrading the road for the passage of light vehicles for the exploration (4 x 4 pickup trucks). The cost of the upgrade was estimated at €$1.5 million and involved bridge rebuilding. SRK allocated US$25 million for general site infrastructure to include the upgrade of the road to bring in large equipment.

18.1.4 Mine Operations and Support Facilities

The mine operations support facilities should include provision for:

- Mine administration and dry building;
- Mine truck workshop and warehouse;
- Truck wash facility;
- Truck fuel facility and ready line; and
- Explosives storage.

The facilities will be placed on high ground at plant site option 1 or 2.

Mine Administration and Dry Building

The mine administration and dry building should be a 35 m x 18 m, single-story, pre-engineered, steel-framed structure with walls and roof to be erected upon a spread footing foundation. The building will provide offices for the mine operations staff, change-house facilities, conference/training facilities, toilets, break room, safety, and first aid.

Mine Truck Workshop

An example of a mine truck workshop building should be sized at 50 m x 20 m, with high bay, pre-engineered, steel-framed structure with walls and roof to be erected upon a spread footing foundation. The building should provide a five-bay maintenance area designed to repair and maintain the mine fleet and other mobile equipment inclusive of CAT 777 haul trucks, loaders, dozers, graders, etc. The electrical room, warehouse and a compressor room should be located adjacent to the maintenance shop.

Truck Wash Facility

A separate truck wash station, equipped with a washing system with a water/oil separator for heavy mining equipment, should be installed outdoors.

Truck Fuel Facility and Equipment Ready Line

The vehicle fueling facility and ready line should be located at the entrance to the plant area adjacent to the main haul road access to the mine pits. The ready line should be located adjacent to the fueling facility and well lighted for 24-hour use.

18.1.5 Process Support Facilities

Adjacent to the plant, the process operations support facilities should include:

- Administration building;
- Laboratory;
- Workshop, warehouse and storage yard; and
- Entry station.

Administration Building and First Aid Facility

The administration building should be located in the plant area and south of the process facilities. The building is sized as a 25 m x 18 m, single-story, pre-engineered, steel-framed structure with walls and roof to be erected upon a spread footing foundation. The building should provide offices for the process operations staff, conference/training facilities, toilets, break room, and safety.

Laboratory

The laboratory building should be approximately 50 m x 30 m, single-story, pre-engineered, steel-framed structure with walls and roof to be erected upon a spread footing foundation. It may be located adjacent to the main process facilities. The laboratory will house sample preparation, assaying, testing facilities along with supporting sample and chemical storage rooms.

18.1.6 Additional Support Facilities

Communications

A radio base station should be provided for plant wide site-to-office communications within a 10 km radius from the communications center for use in the following areas:

- Process plant;
- Plant office;
- Laboratory;
- Mine workshop and offices;
- Central control building; and

This configuration will reduce wiring costs and allow voice messaging integration with e-mail. End-to-end IP video connectivity with business quality transmission will provide video conferencing capabilities.

Satellite Communications

A satellite communications network should be provided for site-to-site communications. The system should include voice/data/video/fax, internet, and VPN services, including bidirectional links.

Satellite phones will be installed at strategic areas for emergency communications.

Fiber Optic

The IT system should be based at the communications building and connected throughout the site by a fiber optic network. The connection between IT devices and end-users will provide high through put, secure, reliable and redundant service for data and voice. The network system should be connected to protocol independent multicasts and business networks through routers with firewalls, which will provide remote access as required. The system should have security and encryption to prevent unauthorized access.

18.1.7 Power Supply and Distribution

Electrical power will be generated on site. Total power for the process plant is roughly estimated at 30 kWh/t, which is typical for this type of plant given the moderate rock hardness. At 12,500 t/d the plant would need to process 565 t/h (at 92% operating availability). This would imply a minimum power generating requirement of 17 MW just for the plant. Given other power needs (surface infrastructure, man-camp, etc.), a 20 MW power generating station has been considered along with two 4 MW generators for backup. The capital cost estimate for the power generating station is based on preliminary quotes from Wartsilla and MAN, which would be provided as a turn-key installation.

18.1.8 Water Supply

At this time, the envisioned water supply system will be constructed to collect run-off waters from the various site areas, which will be directed to a collection/storage system from which it will be pumped to various tanks for use in the mine and plant. A separate water system will be installed near the man-camp for the reverse-osmosis system to produce potable water.

18.2 Tailings Management Area

SRK evaluated four tailings dam areas for storage capacity, liner required and embankment volumes as these are the major cost items as far as capital cost estimation is concerned. The four different TSF options are detailed in Figure 18.2.1.

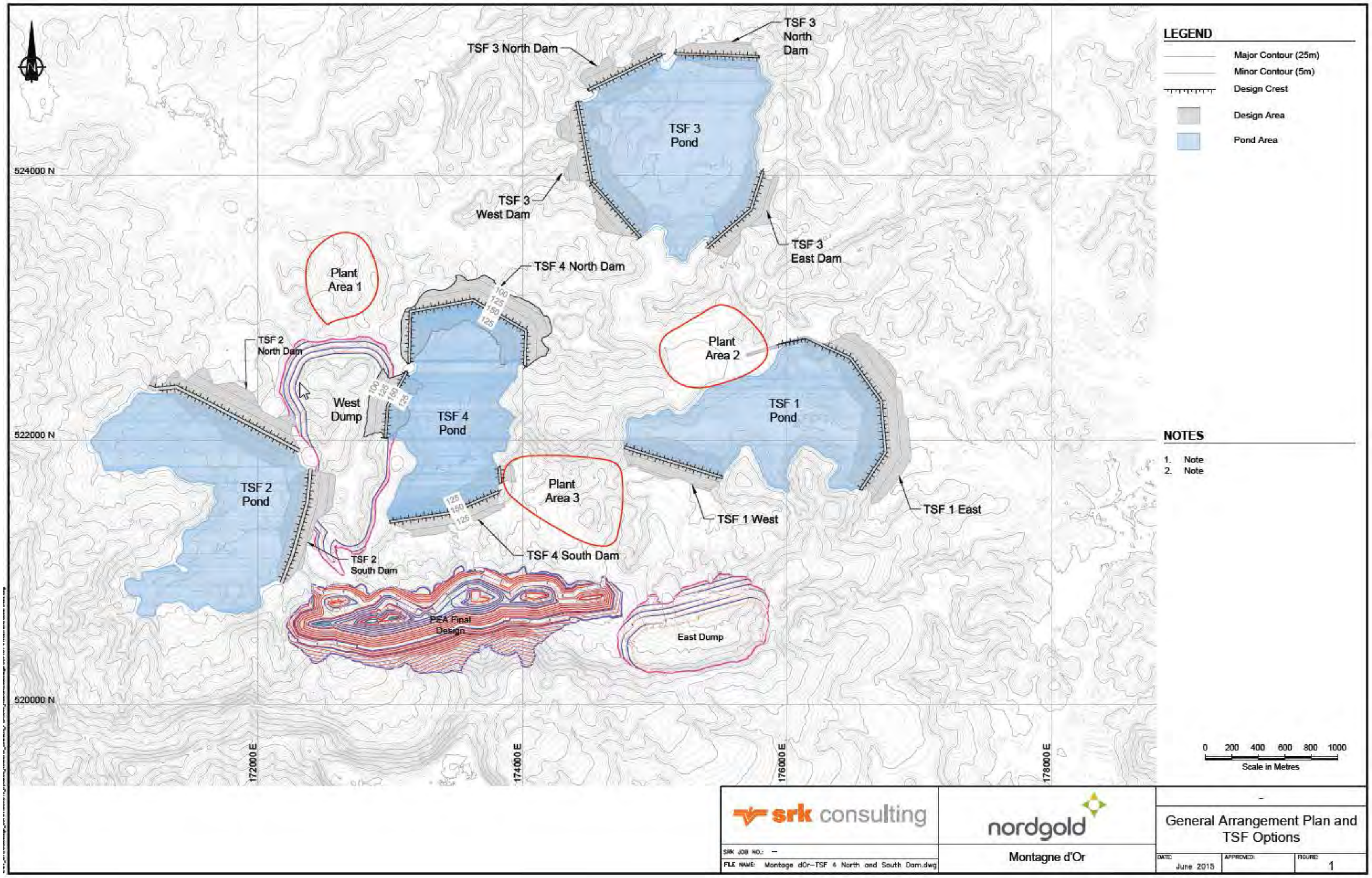


Figure 18.2.1 Tailings Storage Facility Options

Table 18.2.1 details the physical properties of the different tailings dam options. The main reasons TSF 1 and TSF 3 are discounted at this time relates to the extra distance waste rock would need to be hauled compared to TSF 2 and TSF 4.

Table 18.2.1 Tailings Dam Physicals

Description	Unit	TSF 1	TSF 2	TSF 3	TSF 4
Dam crest elevation	m	160	160	155	150
Dam fill volume	m^3	15,282,903	12,367,573	20,064,985	19,997,950
Pond elevation	m	159	159	154	149
Storage volume	m^3	42,160,430	41,105,403	55,675,285	43,167,109
Fill/storage ratio		0.36	0.30	0.36	0.46

Source: SRK, 2015

SRK and Nordgold determined that the most suitable TSF was TSF 4, even though TSF 2 is more efficient from an embankment versus storage capacity point of view. The detrimental attributes to TSF 2 are:

- TSF 2 is mostly but not fully on permitted land;
- No environmental surveys have been conducted at the TSF 2 site, unlike those completed at the TSF 4 or selected site;
- It is likely that the mineralized material continues along strike into the TSF 2 footprint;
- Liner cost is higher than TSF 4 because the basin is shallow; and
- Potential geotechnical interaction between the tailings and open pit.

For these reasons, it was determined that the capital cost estimate for TSF 2 would not be used in the PEA.

The major benefits of TSF 4 are:

- It is centrally located over already disturbed ground therefore delivery of waste to build the embankment has only small incremental cost over transporting waste to a dedicated waste dump;
- The liner costs are less than TSF 2 because the tailings basin is deeper and supported by natural terrain; and
- Central to process plant locations thus reduced pumping costs.

A more detailed image of TS F4 is illustrated in Figure 18.2.2 and as a result of the trade-off exercise TSF 4 is considered the official TSF of the PEA.

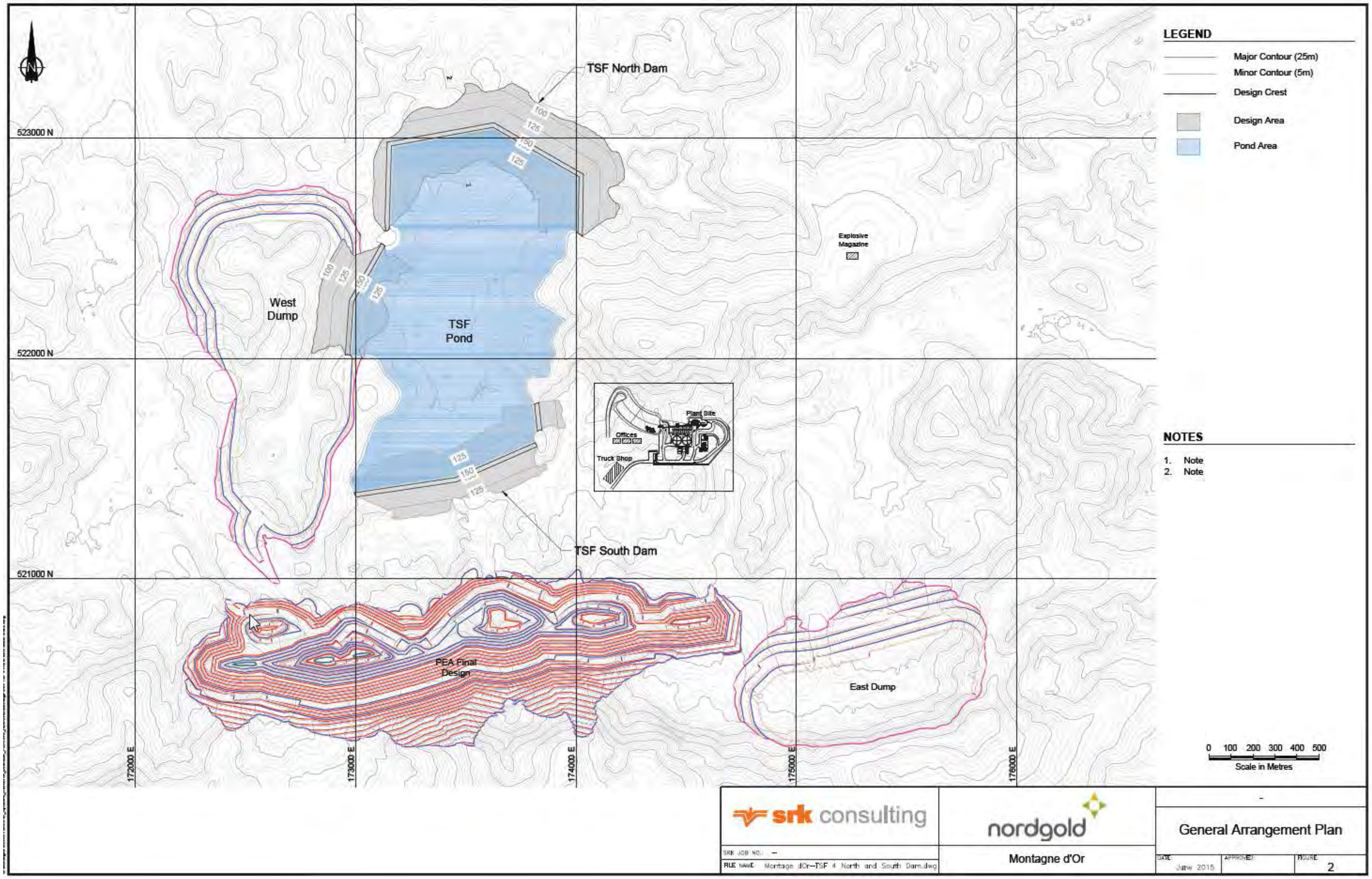


Figure 18.2.2 TSF Tailings Dam and Site Layout

For the purpose of capital cost estimation, TSF 4 and TSF 2 were investigated in detail. Table 18.2.2 shows the capital cost for TSF 4 or TSF that was used in the SRK economic model.

The significant cost drivers are the liner, and the embankment spread and compaction cost. The cost of the liner was estimated at US$10/$m^2$ installed. The reason why there is zero cost for the embankment load haul and dump cost is that the tailings dam wall is a substitute for waste dump material coming from the open pit.

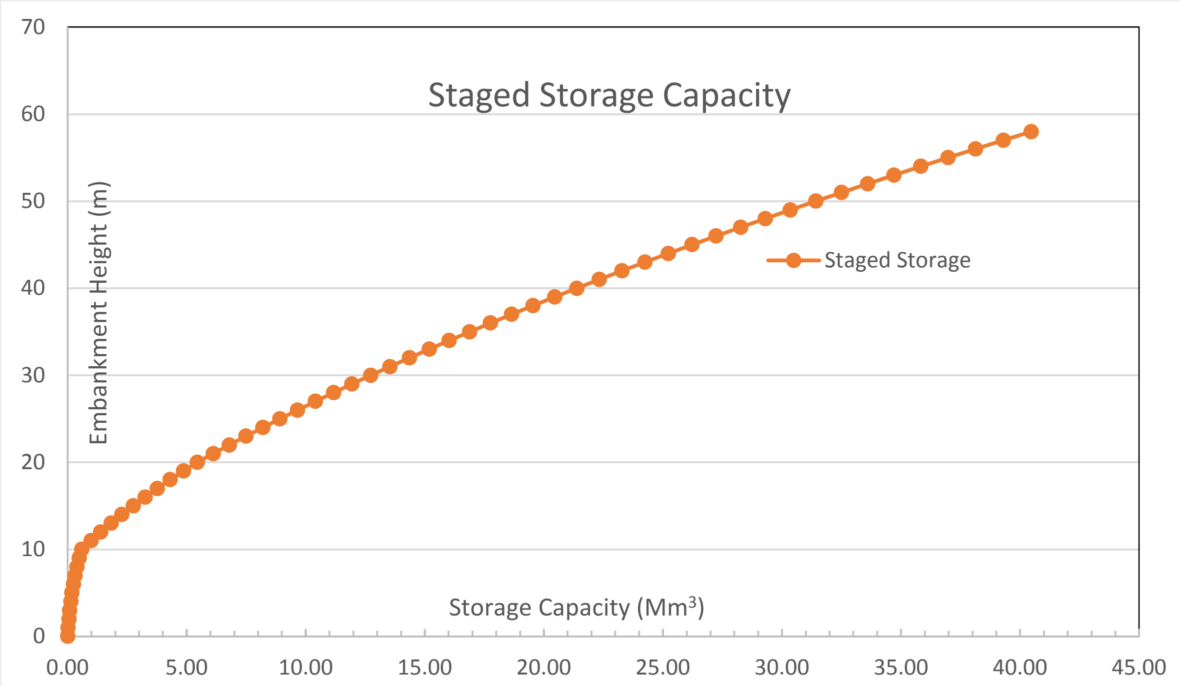
The TSF has been staged in 4 passes after its initial construction during the pre-production period. It should be noted that the tailings will be built using a "downstream" construction method.

Table 18.2.2: TSF Capital and Sustaining Cost Estimate (US$)

Summary		Initial Capital Cost	Sustaining Capital Cost			Total Capital Cost
			Year 2	Year 5	Year 8	
TSF 4	Embankment Clearing	918,064	1,346,530	1,240,494	1,970,916	5,476,003
	Embankment load haul dump	0	0	0	0	0
	Embankment spread and compact	4,915,301	6,662,685	4,170,429	2,500,814	18,249,230
	Embankment transition geotextile	274,183	488,517	466,943	617,533	1,847,175
	Embankment Liner	548,366	977,034	933,885	1,235,065	3,694,350
	Pond Liner	10,372,554	3,787,143	1,675,386	1,743,293	17,578,376
	Tailings & Reclaim Pipeline	72,615	0	0	0	72,615
	Spillway & Instrumentation	150,000	150,000	150,000	150,000	600,000
Diversion	Excavate and riprap	1,953,125	0	0	0	1,953,125
Seepage Collection	Lined ponds	206,025	0	0	0	206,025
Total		**$19,410,233**	**$13,411,909**	**$8,637,137**	**$8,217,620**	**$49,676,898**

Source: SRK, 2015

Figure 18.2.3 illustrates the tailings capacity curves for TSF. Even at full capacity assuming a tailings density of 1.4 t/m^3 the 56 Mt of mill feed requires an embankment height of 58 m. During the FS there is the potential to improve this curve through the optimization of the dam walls.



Source: SRK, 2015

Figure 18.2.3 Embankment Height versus Storage Capacity Curve

18.3 Area Hydrology and Water Balance

The environment for Montagne d'Or is defined by large tropical rainfall events. As part of a site wide water balance, groundwater and surface control program, SRK has investigated the initial water balance for the tailings dam to determine if the system will be net water consuming or net producing. This is important to determine if the Project is likely to require water treatment, not only for the tailings dam water, but also contact waters coming from land disturbance, i.e. waste dump and pit.

Assuming water is diverted around the ultimate tailings dam footprint there are three major sources of water consumption in the closed water system:

- Evaporation—SRK estimated an evaporation rate of 850 mm/y for pan evaporation and 637 mm/y for small water body evaporation estimated from published information (atomos-chem-phys.net);
- Water Bound in Tailings—The calculation is based on tailings density and production rate; and
- Water absorbed by undisturbed ground within the tailings footprint (5% assumed to run-off into dam).

The water added to the system will be from rainfall and process water makeup. At this time it is assumed that clean and dirty contact water can be diverted, and dirty contact water will not be placed in the tailings dam but rather treated elsewhere. Table 18.3.1 details an estimate of a water balance for the tailings dam using capacity curve, evaporation and rainfall information.

Table 18.3.1: Annual Tailings Water Balance by Phase

Description		Starter	Year 2	Year 5	Year 8
Area of Tailings	m^2	141,241	348,399	539,244	842,462
Rainfall	mm/y	3,984	3,984	3,984	3,984
Pan Evaporation	mm/y	850	850	850	850
Shallow Lake Evaporation	mm/y	637.5	637.5	637.5	637.5
Runoff / Rainfall Ratio		0.05	0.05	0.05	0.05
Specific weight Tailings (SG)		2.80	2.80	2.80	2.80
Density deposited Tailings	t/m^3	1.4	1.4	1.4	1.4
Voids in Tailings	m^3/t	0.36	0.36	0.36	0.36
Mineralized Material Production	Mt/y	4.5	4.5	4.5	4.5
Run-on to Tailings	m^2	785,468	578,309	387,464	84,246
Water Contributions					
Precipitation on Tailings	m^3/y	562,703	1,388,022	2,148,349	3,356,369
Evaporation from Tailings	m^3/y	90,041	222,104	343,768	537,070
Run-on to Tailings	m^3/y	156,465	115,199	77,183	16,782
Water Bound in Settled Tails	m^3/y	1,607,143	1,607,143	1,607,143	1,607,143
Other losses	m^3/y	18,250	18,250	18,250	18,250
Estimated Balance TSF Only		-996,266	-344,276	256,371	1,210,688

Source: SRK, 2015

The results suggest that when the tailings dam footprint is smaller, there is the ability of the dam to absorb excess water over and above that being added to the system. This is because the small footprint for the dam is aided by the amount of water bounded in the mineralized material. As the surface area of the tails gets bigger, and the vegetation within the footprint does not absorb a lot of water, the water bound in the tails and lost through evaporation is not big enough to store the entire water landing on the dam footprint through precipitation. This result would imply that the dam will be capable of storing water through the first five years of operation, but after which time, water will need to be discharged to another water storage feature or water treatment plant. This assumes that the tailings dam is not fully lined from day one, and the catchment benefits from natural absorption from vegetation within the dam footprint. When that is lost the storage capability hits a tipping point where the water locked up in mineralized material is unable to account for the precipitation within the liner area and the tailings facility becomes a net positive water dam.

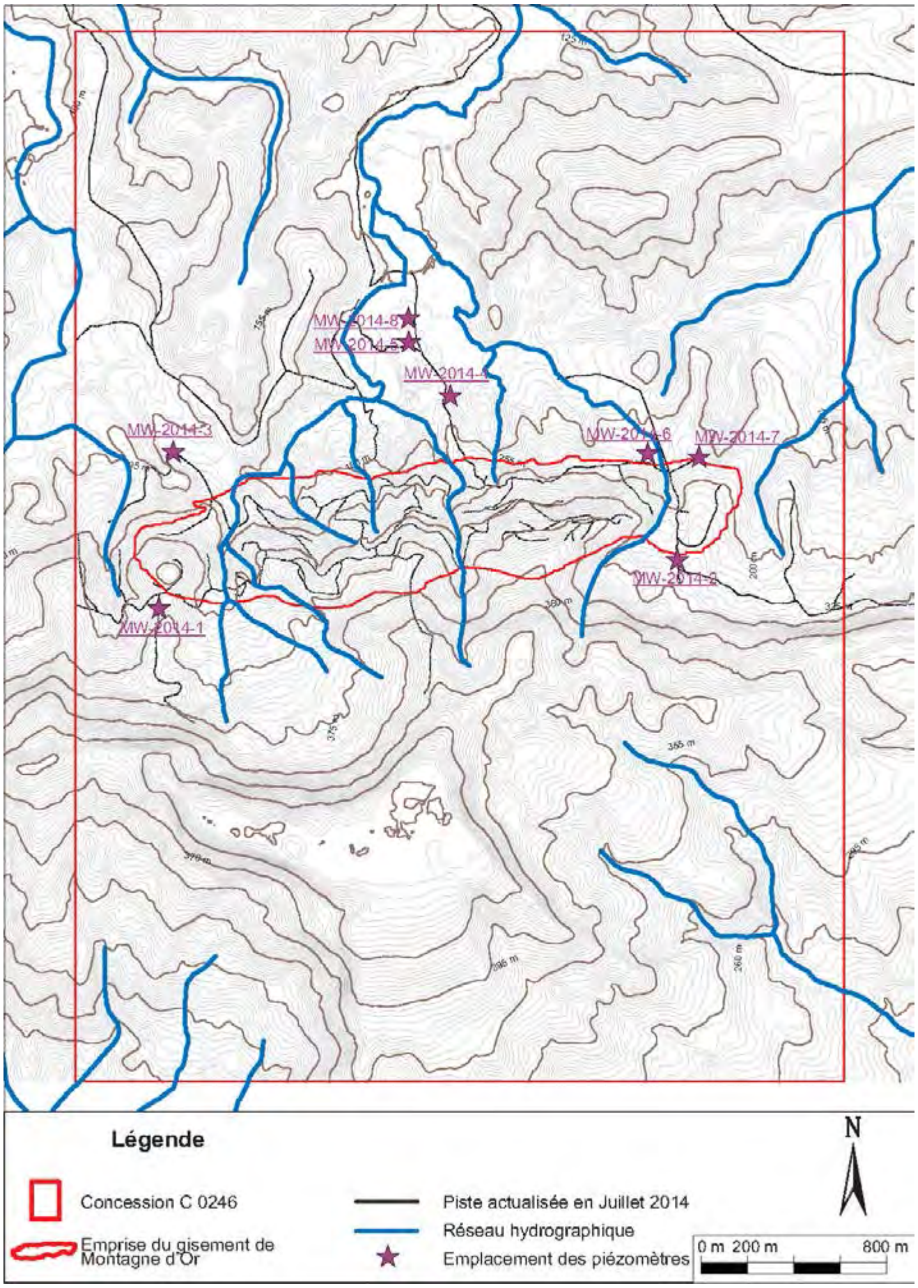
Surface contact water from the pit, waste dump and disturbed land will be separated into clean water (diversion water) and dirty water (collection water). Dirty water will be controlled via a drainage system to a central location where it can either be treated (If needed) or stored in a sump like feature for sediment control. During the FS, the potential turbidity and contaminate levels should be investigated so the final "polishing pond" size can be determined before discharge. If discharge is not permissible then the water may need to be treated independently in the first years of operation or combined when the tailings dam becomes net positive after year 5.

To understand the potential water flows around the pit and potential infrastructure sites, a groundwater reconnaissance program was implemented by WSP as part of the environmental scoping study with the primary focus to understand the circulation in the saprolite/sap-rock around the deposit to be mined. This program's aim was to:

- Install piezometers at eight locations, including two upstream and six immediately downstream from the Project;

- Understand the quantitative aspects of groundwater using four piezometers equipped with water level loggers to monitor the piezometric levels and periodically measure levels in other structures. Results will be correlated with the rainfall and hydrology; and
- Understand the qualitative aspects of groundwater, with two groundwater physico-chemical sampling and analysis campaigns.

Figure 18.3.1 illustrates the well locations around the Montagne d'Or deposit and area disturbed by illegal miners.



Source: WSP, 2015

Figure 18.3.1: WSP Water Sampling Locations

These recent investigations are still being analyzed, treated and interpreted. Thus few results are presented here. However, indications are of low groundwater circulation, localized and sometimes difficult to capture, meaning that 9 of 15 piezometers have water during the rainy season, and 8 of 15 piezometers have water during the dry season. WSP plan to further understand the hydrodynamic situation during the EIA process expected in August of 2015.

As part of the FS it is expected that geotechnical holes will be used for ground water interpretation in and around the open pit.

19 Market Studies and Contracts

No marketing studies or economic analysis have been undertaken for the Project.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Site Visit

SRK's environmental specialist conducted an initial reconnaissance of the Project site from April 1 to 3, 2014. The visit included an overview tour of the proposed mine area, including, but not limited to the location of the open pit (open cut), potential processing areas, and potential tailings storage and disposal areas. Given the early exploration and development phase of the Project, no physical infrastructure of the proposed Project was available for inspection, aside from the existing non-regulated small mining occurring within the proposed pit footprint.

In addition to information gathered during the site visit, SRK was provided a translated copy of the *Preliminary Environmental Report* prepared by WSP Canada Inc. (2015) which provides a preliminary identification of the regulatory elements to which the Montagne d'Or gold Project is likely to be subject, as well as baseline environmental and social information about the Project study area which can be used by the operator during design and development and the regulatory authorities during analysis of potential Project-related impacts.

20.2 Current Liabilities

The Project area is an intermittently active exploration property centered in dense tropical rain forest. Exploration activities require access road and drill pad construction, trenching, water management features, as well as construction of worker camps. Environmental liabilities resulting from previous and ongoing exploration activities are fairly limited due to the high precipitation and rapid natural rehabilitation that occurs in the rainforest. Holders of exploration permits (see below) are required by law to reclaim worked areas, control stormwater and potential sedimentation of downstream surface water resources, and are strictly prohibited from using mercury. These conditions are monitored closely by the government. The previous project owners, and by extension Columbus, negotiated an agreement with French regulatory authorities to dedicate up to €350,000 (US$396,000) to reclamation of exploration disturbances for which it is responsible.

While not the responsibility of Columbus Gold, illegal artisanal placer mining that occurs over much of the Project area has disturbed considerable land area, and continues to impact local surface water resources through increased sedimentation and mercury contamination.

20.3 Required Permits and Status

The *Preliminary Environmental Report* (WSP, 2015) provides a preliminary identification of the regulatory elements to which the Montagne d'Or gold Project is likely to be subject, based on information currently available on the Project. These elements will be confirmed and formalized during preparation of the Project FS.

In 2012, the National Government of France approved new legislation promoting the development of the mining industry French Guiana. The legislation, known as the *Schéma Départemental d'Orientation Minière* (SDOM) was created with the objectives of encouraging economic development of the mining industry in French Guiana while protecting its environment. To accomplish these objectives, the SDOM provides increased security of land tenure, clarifies mineral development guidelines and environmental conditions and restrictions, and assigns lands in French Guiana zones that define limitations on mining activity:

- **Zone 0**: Banned for exploration and mining.
- **Zone 1**: Open to aerial surveys, underground mining authorized subject to conditions.
- **Zone 2**: Open to exploration, underground and open pit mining authorized subject to conditions.
- **Zone 3**: Open to exploration and underground and open pit mining.

Most of the Paul Isnard concession areas, including the Montagne d'Or gold deposit, lie within Zone 2. Some of the conditions to mining in Zone 2 include:

- Demonstration of a viable mineral deposit;
- Completion of an Environmental Impact Study and Reclamation Plan; and
- Possible additional reclamation or environmental investigations, as may be required for the public interest, on or off site.

In addition to the land restrictions presented by the SDOM, the Project is located adjacent to a nature reserve, the Réserve Biologique Domaniale Lucifer Dékou-Dékou, managed by the ONF. Its Management Plan from the ONF is yet to be ratified, so there is little guidance or decisions regarding the use of land and allowable activities within the reserve. The boundaries of this reserve overlap four of the eight Paul Isnard mineral concessions however only one of these concessions is important to the project. Since these concessions already exist, and there has been continued exploration and mining activity in the area for over 100 years, the ONF has agreed to create several zones within the reserve boundaries where mining is permitted. The Montagne d'Or deposit itself is within a zone where open pit mining is permitted and the outer limit of the resource pit shell is located approximately 240 m from the reserve boundary.

20.3.1 Required Permits and Status

French Guiana's mining regime is governed by the legislative and regulatory regime applicable to the French mainland with the exception of certain legal and regulatory provisions which are specific to it in order to take into account particular characteristics and constraints of this overseas territory. Reformation of the Mining Code, however, was proposed in 2012, but has not yet been approved or promulgated. As such, the discussion herewith remains focused on the current permitting requirements. Additional information regarding the proposed reforms is provided later in the text.

French Guiana developed a Departmental Mining Plan in 2011 which "defines the terms and conditions applicable to mining prospection [exploration], as well as the terms of the implementation and exploitation of land mining sites" with a view on economic sustainability as well as environmental protection. The general provisions of the Mining Code provide for two types of mining titles: the exclusive exploration permit ("*permis exclusif de recherche*" or PER) for the exploration phase, and the concession (Concession) for the exploitation phase. A PER grants exclusive rights to carry out exploration activities within a specified exploration area. It is granted for an initial maximum period of five years, but can be renewed twice. A Concession confers on its holder an exclusive right, within the boundaries of such Concession, to explore and exploit the Mineral Resources that it covers. It is assignable and leasable, but cannot be mortgaged, and has an initial maximum term of 50 years and may be subject to successive 25-year renewal periods. Both the issuance of a PER and the granting of a Concession include public disclosure and participation in the permitting process.

In addition, small-scale mining, including most lawful alluvial operations, are carried out through exploitation authorizations ("*autorisation d'exploitation*" or AEX) granted for areas no larger than 1 km2. There are no current AEX operations within the Paul Isnard Project area.

The Paul Isnard Project does not currently include any PER. Instead, the Project is comprised of eight (8) mining concessions covering approximately 135 km^2. The mining concessions, combined with appropriate permits, allow large-scale mine operations and are valid until December 31, 2018 with potential renewal for a maximum of 25 years conditional upon a number of conditions, not the least of which is proving economic viability.

The Project is located entirely within Concession No 215 - C02/46, held by SOTRAPMAG, a subsidiary of COLUMBUS GOLD. This concession was granted on May 21, 1946 (J.O. of June 1, 1946) to S.E.E.M.I., then ceded to SOTRAPMAG by the Decree of December 27, 1995 (J.O. of December 29, 1995) for an unlimited term. However, as the Mining Code's new article L. 144-4 provides for the expiration of unlimited term mining concessions on 31 December 2018, this concession will indeed expire on December 31, 2018. A first 25-year extension will be granted (through a simple mail-in request to the minister in charge of mines and to French Guiana's DEAL) if, at the time the concession extension request is submitted, that is December 31, 2016, SOTRAPMAG is able to demonstrate that legal gold production is being carried out within the concession. Otherwise, a new concession application would need to be submitted to the minister in charge of mines and will be open to competition.

The Project does include a pending application for an exclusive exploitation permit ("*permis d'exploitation*" or PEX) covering an additional 14.4 km^2 outside of the concession areas. The PEX, combined with appropriate operating permits, also provides for medium- to large-scale mine operations, and is granted for five years with two potential and maximum renewals of five years each. The Paul Isnard mining concessions, and the pending PEX, require quarterly reporting to the State but carry no defined financial commitments for maintenance.

20.3.2 Facilities Classified for Environmental Protection (ICPE)

The Environment Code has specific regulations for facilities owned or operated by any public or private natural or legal person, which may present dangers or inconveniences for neighbors, health, safety, public hygiene or the environment. These *Facilities Classified for Environmental Protection*, or ICPE, are subject to authorization, registration or declaration depending on the extent of the dangers or inconveniences caused by their operation. Included in these are:

- Ore processing-related infrastructure,
- Energy production infrastructure,
- Use of explosives, and
- Ancillary activities at the base camp (e.g., hydrocarbon storage and distribution, workshops, air conditioning systems, sawmills, etc.).

20.3.3 Restoration of the Access Road from the Croisée D'apatou

The Project to restore the Montagne d'Or site's access road from the Croisée d'Apatou will be subject to an impact assessment and public enquiry, since these activities could lead to changes in the long and cross profiles of the minor beds of creeks crossed by the road, or the diversion of these creeks.

20.3.4 Law on Water and Aquatic Environments

Various activities necessary for the development of the Project are likely to be subject to the Environment Code and its requirements, including:

- Development of process and potable water supplies;
- Stormwater which contacts mining facilities;
- Tailings management facilities;
- Creek crossing structures;
- Diversion of natural drainages;
- Facilities located in designated flood zones;
- Process water ponds; and
- Mining infrastructure.

20.3.5 European Directives

Through its association as a Department of France, the Project will likely be subject to the European Directive 2010/75/EU on industrial emissions, or IED, which was established for environmental protection through the pollution prevention. Its guiding principles are:

- The use of BATs (Best Available Techniques) for the subject activities;
- The periodic review of the authorization conditions; and
- The restoration of the site to a state at least equivalent to that described in a "Baseline Report" which describes the state of the soil and groundwater prior to commissioning.

The activities covered by the IED's were introduced into the ICPEs. The directive also gives a list of criteria to be taken into consideration for determining the BATs. The BATs are compiled in reference documents (BREFs), which are produced by the European Commission's European Integrated Pollution Prevention and Control (IPPC) Bureau, or EIPPCB. The Project could be covered by the available BREFs on:

- ferrous metals processing (December 2001 BREF); and/or
- the management of tailings and waste rock in mining activities (January 2009 BREF).

The IED also introduces the requirement to submit a Baseline Report describing the state of the soil and groundwater prior to commissioning of the mining Project. This report is to be used for reference purpose during final closure.

The Project will likely be covered by the ICPE Section 3250 – a) (production of non-ferrous crude metals from ore, concentrates or secondary raw materials by metallurgical, chemical or electrolytic processes). It will therefore be subject to the IED and require a Baseline Report.

20.3.6 Mine Code Reformation

The original proposal and legislation for reformation of the Mining Code, announced in 2012, failed to garner sufficient support for passage late last year. However, that legislation is currently being revisited, and is anticipated to pass, possibly by the end of this year. While the proposals maintain much of the "French mining model" which is based on the ownership of the subsoil by the State (beneath 30 m) and the granting of permits for the exploration or exploitation of Mineral Resources, the new legislation is likely focus on the following areas for change:

- Increased environmental protection;
- Improved worker safety and public safety;
- Protection of mining operators legal position and tenures;
- Simplification of administrative procedures; and
- Inclusion and strengthening of public participation and transparency in the permitting process.

The draft legislation also proposes modifications to the current tax structure, though no specifics are currently available.

20.4 Environmental Study Results

The *Preliminary Environmental Report* (WSP, 2015) provides an overview of the environmental and socio-economic issues for the Project, as well as a preliminary indication of the positive and potentially negative impacts associated with the planned operation, which is intended to provide direction for the continuing environmental assessment process, and guide the environmental authorities with the information required to determine the range of information and degree of detail needed in the formal impact assessment. The study area, as defined for the environmental studies purpose, includes concession n° 215 - C02/46, the access trail from Saint-Laurent-du-Maroni, and a 500 m wide buffer zone on each side of the trail. The following are brief summaries of some of the environmental and social issues presented in the report by WSP (2015).

20.4.1 Air Quality

The field program to establish baseline air quality at the Project site took place during the dry season, from 9 to 15 October 2014. While the precise data on air quality are not yet analyzed, WSP (2015) concludes that the overall air quality is good, given the lack of human activity in the area and the dense forest cover. As a result, the sensitivity regarding air quality will likely be high, especially since the Lucifer Dékou-Dékou Integral Biological Reserve including the Dékou-Dékou massif, to the south of the Project, and the Lucifer massif, to the north of the Project, must be preserved.

The Project will be subject to the guidance and recommendations on the use of Best Available Techniques entitled « Non-ferrous Metals Industry » (European Commission, 2009). This document lists the best practices in force regarding the collection and depollution of precious metal treatment processes.

20.4.2 Cultural and Archeological Resources

The terms-of-reference for conducting the archaeological baseline program (including the field investigation) was part of the prospecting authorization by the DAC-SA, Order No. 17, of August 7, 2014. Only two Native American occupation sites were identified in the field, both yielding few artifacts. More modern and contemporary archeology in the area includes artifacts generally associated with past mining activities, dating back to ca. 1873. Several miners' villages were located, but they are completely degraded by illegal gold mining with scattered and broken equipment. Two cemeteries were located at Paul Isnard and Enfin. Overall, the archeological sensitivity within the study area is considered to be low.

20.4.3 Biological Reserves and Resources

Baseline data on the fauna, flora and habitat covering the Project area and presented in by WSP (2015) were obtained from a review of the published literature, existing databases, and field inventories conducted in 2014. These inventories concentrated on terrestrial and aquatic faunal taxa mostly in the vicinity of the mining site and Camp Citron, and were timed to coincide with wet season (May 28 to June 3) and dry season (August 7 to August 12) conditions at the site.

The Lucifer and Dékou-Dékou massifs are home to two floral assemblages rare in French Guiana: the sub-montaneous forest on lateritic bauxite hardpan, and the forest on 400 to 500 m slopes. They shelter some fifty floral heritage species and three nationally-protected species. This heritage value led to the creation in 2012 of the Lucifer and Dékou-Dékou Integral Biological Reserve, the first such reserve in French Guiana and the largest in France. Within the reserve, any direct human intervention that could modify the functioning of the ecosystem is prohibited. The only authorized sylvicultural measures are those eliminating exotic or invasive species and the securing of trails and roads bordering or crossing the reserve

However, there are limited exceptions. While the Project itself is located in portions of a managed biological reserve, mining activity is permitted under certain conditions. This exception was established to take into account historic exploration and exploitation of gold resources in the area, as well as the presence of potentially significant mineral deposits at the foot of the Dékou-Dékou massif.

The field inventories conducted in 2014 in the study area found 467 species of plants and 370 species of terrestrial vertebrates. The aquatic fauna surveyed comprised 52 families of macro invertebrates and 41 species of fish.

The review of existing data added 68 more plant species found in the study area, bringing the total potential number of special to 535. The highest diversities were found in the steep slope old-growth and dense evergreen forests. The richest community of amphibians, almost half the number of species observed, was found in the alluvial mining pits and their interfaces with the second-growth forest. A total of 226 species of birds were recorded in 2014. This is about a third of all species known in French Guiana (665). Only 27 non-volant mammalian species were identified during the field surveys.

As part of the biological baseline data collection program, fish tissue samples were analyzed for mercury content using an AMA 254 spectrophotometric absorption mercury analyzer. A total of 245 specimens were sampled across several locations. Stations located upstream and within the deposit area did not produce any specimens with concentrations greater than the World Health Organization (WHO) limit of 0.5 µg Hg/g. In contrast, the pit stations yielded 39 samples above the WHO limit, as would be expected in locations where illegal artisanal mining which uses mercury was occurring.

20.4.4 Threatened, Endangered, and Special Status Species

Article L. 411-1 of the Environmental Code strictly protects wild species of plants and animals listed by Ministerial Order. These species cannot be captured, transported, intentionally disturbed or commercially exploited. These prohibitions can extend to the destruction, degradation and alteration of the habitats of these protected species.

20.4.5 Land Use

In the Project area, most land (including the trail between Saint-Laurent-du-Maroni and Citron Camp) consists of wet lowlands forest. Near Saint-Laurent-du-Maroni, slash-and-burn farming is performed on small plots along the road toward the Croisée d'Apatou. This road crosses the State's private forest. This forest, and the trail up to Croisée d'Apatou, is managed by the National Forestry Office. A few forest exploitation sites, a sawmill and at least three laterite quarries are located near the trail.

20.4.6 Hydrogeology (Groundwater)

A baseline groundwater reconnaissance program was conducted at the site as part of the *Preliminary Environmental Report* (WSP, 2015). A network of 15 piezometers was installed around the Project area, with the following objectives:

- Periodic monitoring of piezometric levels at all piezometers and continuous monitoring at 4 of them, with 4 water level loggers recording at 30-minute intervals, over a long enough period (generally for 1 complete hydrogeological cycle, that is 1 year). This short time step will allow the synergy between the groundwater and precipitation to be assessed. Continuous monitoring will provide information on inflexion points over time, commonly representing the so-called "low- and high-water" periods occurring during monitoring;
- Determining the hydrodynamic parameters of the massif, including the physical values: permeability, porosity, storage coefficient;
- The physico-chemical characterization and the search for contaminants at the baseline (initial) state in the groundwater.

The preliminary results indicate that there is no alluvial aquifer potential in the vicinity of the Montagne d'Or deposit. Bedrock groundwater movement appears to be fault controlled. Within the area of the mineral deposit, groundwater occurs in the saprolite/sap-rock (extension over nearly the entire Project, at a depth of 5 to 35 m) and to a lesser extent by localized deep groundwater flow.

Early results from the groundwater quality monitoring program showed that the waters are slightly acidic with low major element concentrations, a bicarbonate calcic profile, and no contamination (i.e., no elevated concentrations of regulated constituents).

20.4.7 Geochemistry

Preliminary geochemical characterization of representative lithologies (approximately 30 waste rock and mineralized samples from sections of exploration drill core) of the Montagne d'Or waste rock and low-grade mineralized material has been conducted to assess the potential for acid rock drainage and metal leaching (ARDML). The samples were tested at Inspectorate Exploration & Mining Services Ltd. (IEMS) laboratory in Vancouver. In addition, six tailings samples from the metallurgical testing program were also subjected to geochemical analysis.

Waste Rock

Static testing was conducted in accordance with French standards (NF EN 15875). The results suggest that 17% of the samples have no acid-generation potential and that 12% of samples are potentially acid generating. The remainder of the samples were indeterminate for acid generating potential. As a result, these material types will have to be categorized as potentially acid generating and managed as such.

A single column kinetic test was undertaken on a composite sample taken from the same 30 waste rock samples used in the static testing program. This test is ongoing, and will require many more weeks before the data can be analyzed and conclusions drawn with respect to the longer-term acid generating potential of the Montagne d'Or waste rock.

Leaching tests were also conducted on the 30 waste rock samples per standard NF EN 12457-2, "Compliance test for leaching of granular waste materials and sludges". The tests showed that four samples (or 13%) showed potential for leaching arsenic (As), copper (Cu) or zinc (Zn). The samples showing leaching potential have either acid generation potential or uncertain potential. In light of these results, the potential for leaching metals remains a potential concern at this stage, and will need to be considered during design and development of the mine.

Low-Grade Mineralized Material

Tests conducted on 30 low-grade samples showed that 23 (or 77%) showed acid generation potential, while seven other samples showed uncertain potential. Six of the 30 samples (or 20%) showed potential for leaching mercury (one sample) or copper (five samples). The low-grade mineral samples tested are thus classified as potentially acid generating.

Tailings

Of the six tailings samples analyzed, four (or 67%) are classified as potentially acid generating. The two other samples show uncertain potential. None of the samples show elevated metal-leaching potential. The tailings samples analyzed should nonetheless be considered as being potentially acid generating.

20.5 Environmental and Social Management Planning

WSP (2015) includes recommendations on avoidance strategies, mitigation alternatives, compensation, and monitoring measured in order to ensure compliance with the respective regulatory frameworks for the Project and the environmental resources. These actions and activities are typically detailed in Environmental Management Plans (EMPs) and Social Management Plans (SMPs).

It is expected that the measures applicable to the Project will be validated and more clearly defined alongside the Project's technical design process. The development of additional environmental management and effects monitoring measures for the Project is to be expected as part of the environmental assessment process. The measures shall be proportionate to the issue and magnitude of the anticipated impacts.

The principal areas of potential impact, and thus the focus of Project environmental and social management planning are likely to include:

- Soil stability,
- Soil conservation,
- Pollution prevention,
- Biodiversity,
- Economic development,
- Natural resources,
- Archeology, and

- Safety and crime.

20.6 Reclamation and Closure

Upon final closure, the operator is required to provide an assessment of the final soil and groundwater conditions in comparison to the previously developed Baseline Report. The operator is required to restore the site to a state that is, at a minimum, similar to that described in the Baseline Report (articles L. 515-30 and R. 515-75 of the Environment Code). This requirement is in addition to those regarding the restoration for the selected future land use (article L. 512-6-1 of the Environment Code). For new facilities, this report is part of the authorization request.

Given the current lack of mine design information, the costs associated with closure of the Project have been estimated at approximately US$25 million based on similar nature and extent of the operations to projects previously evaluated by SRK. This number will be refined using actual mine designs and country-specific costing rates during development of the Project feasibility study.

20.7 Socio-Economics

Covering an area of 83,846 km², French Guiana has 22 towns in 4 communities of communes and 19 cantons. The closest community to the Project site, the town of Saint-Laurent-du-Maroni, has a Local Urbanism Plan (PLU), approved on 8 October 2013, which contains provisions affecting the development of mining in the region, including:

- develop a higher education sector with the creation of a training center for higher education;
- enable the development of mining in a manner that is consistent with the protection of the natural environment requirements, the preservation of agricultural and forest areas and remarkable ecological environments;
- reconcile the imperatives of urban and economic development and the preservation of the natural environment, heritage and identity, protecting specific areas harboring endemic species such as the ZNIEFF Dékou-Dékou (high points) and Lucifer massifs (high points) and special and remarkable landscapes participating in the area's identity, including Chutes et crique Voltaire; and
- qualify the economic river entrance to the city, for the development of river frontage at ports.

Moreover, the Project sector is classified as an Nf zone (natural protected area due to the presence of wooded areas where the forest’s vocation is recognized) under the PLU; the regulation authorizes the extraction of materials subject to compatibility with the Departmental Mining Plan (SDOM). The latter classifies Montagne d’Or as a zone where mining activity is permitted but under constraints, given the environmental sensitivity.

20.7.1 Gold Mining

Gold mining is fairly well developed in French Guiana, with 42 mining claims and 52 operating licenses on record in 2013. For 2013, French Guiana reported the production of 1.3 t of gold, generating a regional mining tax of €550,000, a departmental and communal fee of €200,000, and a fee to ONF around €200 000. However, the industry continues to be plagued by illegal artisanal mining, which, according to the ONF, consisted of 774 sites in 2013, for an estimated annual production of 10 to 12 t of gold.

20.7.2 Bushinengues

In French Guiana and neighboring Suriname, *Bushinengues* (meaning 'people of the forest') are identified, in part, as descendants of former African slaves who escaped in the late seventeenth and early eighteenth centuries from the Dutch plantations in what is now known as Suriname. Their communities were the result of a growing movement amongst slaves to escape enslavement and establish independent communities in the forest. While a significant portion remained in Suriname, the majority fled to French Guiana.

The Aluku is one of the *Bushinengues* ethnic groups in French Guiana, who, toward the end of the eighteenth century, settled alongside the riverbanks of Lawa Maroni, which now forms the border between French Guiana and Suriname. There were at least two other groups of escaped Africans in the area, Saramaka people and the Ndyuka people, who eventually assimilated with the Aluku to form a new ethnic group.

In the late eighteenth century, the Aluku occupied the region of Saint-Laurent-du-Maroni, Apatou, Grand-Santi; the largest piece of the territory still occupied is called Fochi-ké (First Cry), better known as Aluku, located in the region of Maripasoula, consisting of the municipalities and city of Maripasoula and the capital city of Papaïchton, and the traditional villages of Kormontibo, Assissi, Loca, Tabiki, and Agoodé, in French Guiana, as well as the Cottica, in Suriname. There is also a very large Aluku population in Saint-Laurent-du-Maroni, Cayenne, Matoury, and Kourou.

20.8 Environmental / Social Issues and Impacts

20.8.1 Principal Issues

WSP (2015) summarizes the main issues and concerns expressed by stakeholders during a first series of consultations which took place in September 2014. The purpose of identifying the issues and concerns is:

1. to guide the Project's continuing environmental assessment process, specifying the factors which will require specific attention; and
2. to focus the Project early on in the design process, specifying the environmental and social considerations needing to be taken into consideration to ensure compliance with the regulatory framework and to avoid significant effects on the natural or human environments.

While WSP (2015) classifies these issues and concerns according to three levels of significance, only those identified as Level 1 are presented herein. Level 1 is given to issues which are subject to standards or regulatory aspects, which are deemed important for the Project's acceptance by stakeholders, and which could generate significant environmental or social impacts.

- Biodiversity and natural spaces:
  - Integrity of the Lucifer Dékou-Dékou / Integral Biological Reserve;
  - Protection of flora and fauna and quality of biological inventories;
- Stakeholder consultations:
  - Proactive and transparent communication;
- Economic development:
  - Local and regional jobs and economic spinoffs;
  - Supply of energy;

- Training of qualified local workforce;
- Fight against illegal gold mining:
  - Contribution to the fight against illegal gold mining;
- Pollution prevention:
  - Sound environmental management;
  - Prevention of pollution and industrial risks, including those related to the eventual use of cyanide;
- Protection of watercourses and catchments:
  - Protection of catchments (Mana, Sparouine);
- Safety and crime:
  - Controlling traffic on the Paul Isnard track;
  - Securing the mine site; and
  - Workplace health and safety.

Level 2 issues and concerns have no applicable regulations, but are deemed important for certain stakeholders or could produce significant environmental or social impacts, while Level 3 is used to classify issues for which it is generally desirable to avoid negative environmental or social impacts.

20.8.2 Project Advantages

The Project is likely to generate positive effects which must also be taken into account during the environmental assessment and design processes. In general, the main advantages of natural resource projects are the creation of direct and indirect jobs, the stimulation of companies who supply products and services, and increased fiscal benefits related to economic stimulus and royalties. The stakeholders consulted in September 2014 identified additional opportunities related to the Project's specific situation:

- Fight against illegal gold mining in the region;
- Reduce gold pillaging from French Guiana;
- Stop discharge of mercury into the environment (its use has been banned since 2006);
- Stop damage to the Lucifer Dékou-Dékou Integral Biological Reserve, to the bottom of valleys and to creeks, as well as wildlife poaching by illegal miners.
- Improve development of mining industry in French Guiana;
- Increase revenue to the region;
- Train qualified workers in various technical and professional sectors; and
- Restoring degraded sites in the Lucifer Dékou-Dékou Integral Biological Reserve.

20.9 International Standards and Guidelines

Even though French Guiana (through its connection with France) is a Designated Country with respect to the Equator Principles, Nordgold has committed to ensuring that Montagne d'Or is in compliance with international standards and guidelines, to the extent practicable, given the potential for international investment in the Project. Designated Countries are those countries deemed to have robust environmental and social governance, legislation systems and institutional capacity designed to protect their people and the natural environment.

Potentially relevant international policies and/or guidelines for which the Project is likely to maintain compliance with include, but are not necessarily limited to:

- Equator Principles risk management framework for determining, assessing and managing environmental and social risk in projects;
- International Finance Corporation (Performance Standards) (IFC – PS) – social and environmental management planning;
- World Bank Guidelines (Operational Policies and Environmental Guidelines);
- Vienna Convention for the Protection of the Ozone Layer;
- Montreal Protocol on Substances that Deplete the Ozone Layer;
- Basel Convention on the Control of Trans-boundary Movements of Hazardous Wastes and their Disposal;
- Rotterdam Convention on the Prior Informed Consent Procedure for Certain Hazardous Chemicals and Pesticides in International Trade
- Stockholm Convention on Persistent Organic Pollutants;
- United Nations Climate Convention and the Kyoto Protocol; and
- Grenelle Environment Round Table of 2007 and the Grenelle Law II.

Table 20.9.1 provides a brief assessment of the approach to compliance anticipated for Montagne d’Or with respect to the IFC Performance Standards, even though the French Guiana is a Designated Country.

Table 20.9.1: IFC Performance Standard vs. Compliance Approach

IFC Performance Standard (PS)	Summary of Requirements	Project Compliance
PS1: Assessment and Management of Environmental and Social Risks and Impacts	Development of an ESMS appropriate to the nature and scale of the Project which includes a policy, identification of risks and impacts, management programs, organizational capacity and competency, emergency preparedness and response, stakeholder engagement, monitoring and review.	Project will be subject to environmental impact assessment and environmental management requirements at various stages of the of the permitting process
PS2: Labor and Working Conditions	Identification of risks, impacts and management requirements associated with working conditions and terms of employment, non-discrimination and equal opportunity, retrenchment, grievance procedures, child labor, forced labor, occupational health and safety, third party workers and the supply chain.	Project will be governed by French and EU statutes and regulations, as well as local requirements
PS3: Resource Efficiency and Pollution Prevention	Promotes technically and financially feasible options to address resource efficiency (including greenhouse gas production and water consumption) and pollution prevention (with respect to wastes, hazardous materials management and pesticide use) across the Project life-cycle.	Project will be governed by French and EU statutes and regulations, as well as local requirements, and some international standards (e.g., WHO, etc.)
PS4: Community Health, Safety and Security	Evaluation of risks and impacts to the health and safety of Project-affected communities over the Project life cycle. Issues to be considered include infrastructure and equipment design and safety, hazardous materials management, ecosystem services, community exposure to disease, emergency preparedness and response, and management of security personnel.	Project will be governed by French and EU statutes and regulations, as well as local requirements, and some international standards (e.g., WHO, etc.)
PS5: Land Acquisition and Involuntary Resettlement	Applies to physical and or economic displacement resulting from Project acquisition of land rights or land use rights through expropriation, compulsory procedures, or negotiated settlements that if fail result in compulsory procedures. This PS also applies to Project situations requiring eviction of people occupying land without formal, traditional or recognizable usage rights and situations involving involuntary restrictions on land use or use of natural resources.	Due to its remoteness and location, there will be no involuntary resettlement associated with the Project.
PS6: Biodiversity Conservation and Sustainable Management of Living Natural Resources	Identification of risks and impacts on biodiversity and ecosystem services, especially focusing on habitat loss, degradation and fragmentation, invasive alien species, overexploitation, hydrological changes, nutrient loading and pollution. Guidance measures are dependent on type of habitat present (i.e. modified, natural or critical). Where a project is likely to adversely impact ecosystem service, a systematic review to identify priority ecosystem services is required.	Wetland and riparian resources impacted by the Project will be mitigated in accordance with Environmental Code and international biodiversity agreements
PS7: Indigenous Peoples	Avoidance of adverse impacts on indigenous peoples and active engagement with the affected communities. Free, prior and informed consent (FPIC) of affected communities of indigenous peoples is required for projects with potential impacts to lands and natural resources subject to traditional ownership or customary use, relocation of indigenous peoples from such lands, and impacts to critical cultural heritage.	There are no classifiable indigenous peoples in the area of the Project, though there are some *Bushinengues* and aboriginal populations in the region
PS8: Cultural Heritage	Promotes protection of cultural heritage in Project design and execution including implementation of chance find procedures, consultation, and community access and mitigation hierarchy. Critical cultural heritage should not be removed, significantly altered or damaged.	Operator will work with the Directorate of Cultural Affairs (DCA) and/or Regional Archaeology Department (SRA) to ensure that no cultural heritage is impacted by the Project

21 Capital and Operating Costs

21.1 Capital Cost Estimates

LoM capital costs totaling US$476 million are summarized in Table 21.1.1. Approximately 12% contingency has been applied to capital items, which is appropriate for a PEA level of analysis in SRK opinion. The initial capital is estimated to US$366 million which is the estimated investment to construct the Project that will produce approximately 265 koz/y during the first 11 years of the operation when little stockpile material is fed to the mill.

Table 21.1.1: Life-of-Mine Capital Costs (US$000's)

Description	Initial	Sustaining	Post Closure	LoM
Pre-Stripping	27,027	152,692		179,719
Open Pit Mining	53,513	33,143		86,656
Processing	136,741	0		136,741
Tailings	19,410	30,267		49,677
Infrastructure	70,500	0		70,500
Owner's Cost	14,875	0		14,875
Reclamation/Closure/Equipment Salvage	0	0	25,000	25,000
Subtotal	**322,066**	**216,102**	**25,000**	**563,168**
Contingency (14% of Initial capital cost)	44,360	0	0	**44,360**
Total Capital	**366,425**	**216,102**	**25,000**	**607,527**

Source: SRK, 2015

21.1.1 Mine

The estimated cost of mine equipment and timing of purchases are shown in Tables 21.1.1.1 and 21.1.1.2. Mine capital equipment costs were obtained from recent cost models and handbooks. No sustaining capital was estimated.

Table 21.1.1.1: Open Pit Mining Capital Costs (US$000's)

Description	Initial	Sustaining	LoM
Drilling	2,664	3,374	6,038
Loading	13,521	6,386	19,907
Hauling	29,092	17,026	46,118
Roads & Dumps	8,236	6,357	14,593
Total Open Pit Mining	**$53,513**	**$33,143**	**$86,656**

Source: SRK, 2015

Table 21.1.1.2: Open Pit Mine Capital Costs

Equipment	Unit Cost (US$)	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Drilling																		
PowerROC D55	888,000	1	2	-	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Loading																		
Cat 374	1,079,000	2	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cat 988	912,700	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Backhoe 12.0 m^3 Bucket	3,428,000	1	2	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Hauling																		
ADT40	710,000	13	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
CAT 777	1,368,000	3	11	2	-	5	-	-	-	-	-	-	-	-	-	-	-	-
Roads & Dumps																		
Cat D9	990,080	2	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Spare Cat D9	990,080	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-
16 m Grader	990,080	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Truck	844,900	2	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Backhoe 1.76 m^3 Bucket	398,500	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Source: SRK, 2015

21.1.2 Process Capital Costs

SRK prepared capital cost estimates for a production rate of 12,500 t/d flotation and CIP process plant to produce an average of 265 koz/y over the first 11 years of the operation. Table 21.1.2.1 present the estimate of capital costs for mineral processing with 20% contingency added to initial capital. It was considered that the initial capital will be spent during the two years prior to the startup of the process plant. No sustaining capital was estimated.

Table 21.1.2.1: Process Plant Capital Costs (US$000's)

Description	Initial	LoM
Comminution	22,350	42,350
CIP Leaching	60,240	30,240
Solid-Liquid Separation	4,986	8,986
General	4,481	10,480
Engineering/Management	13,129	13,129
Subtotal Area	**$105,185**	**$105,185**
30% Capital Cost Adjustment	31,556	31,556
Subtotal Process Plant	**$136,741**	**$136,741**
20% Contingency	27,400	27,400
Total Process Plant	**164,089**	**164,089**

Source: SRK, 2015

SRK prepared capital cost estimates for the TSF, which show a 15% contingency to initial capital (Table 21.2.2.2).

Table 21.1.2.2: Tailings Storage Facility Capital Costs (US$000's)

Description	Initial	Sustaining	LoM
TSF	19,410	30,267	49,677
Subtotal	**$19,410**	**$0**	**$49,677**
15% Contingency	2,912	0	2,912
Total Tailings	**$22,322**	**$30,267**	**$52,588**

Source: SRK, 2015

21.1.3 Infrastructure Capital Costs

Table 21.1.3.1 presents infrastructure capital costs with 20% contingency added to initial capital. No sustaining capital was estimated.

Table 21.1.3.1: Infrastructure Capital Costs (US$000's)

Description	Initial
HFO/Palm Oil Power Generation (28 MW Nominal)	33,000
Water Treatment Plant	12,500
All Other Infrastructure	25,000
Subtotal	**$70,500**
20% Contingency	14,100
Total Infrastructure	**$84,600**

Source: SRK, 2015

21.1.4 Other Capital Costs

Owner's cost and closure/reclamation costs are presented in Tables 21.1.4.1 and 21.1.4.2. No social costs have been estimated for the Project at this time.

Table 21.1.4.1: Owner's Costs (US$000's)

Description	Initial
Owner's Costs (5% of Direct + Indirect Capital Cost)	14,875
Total Owner's Cost	**$14,875**

Source: SRK, 2015

Table 21.1.4.2: Closure/Reclamation Capital Costs (US$000's)

Description	Post Closure
Mine Closure/Reclamation	25,000
Total Closure/Reclamation/Salvage	**$25,000**

Source: SRK, 2015

21.2 Operating Cost Estimates

Operating costs for both mine and plant consider the following:

- 365 days of operation per year;
- 24 hours of operation per day; and

Table 21.2.1 presents LoM operating costs

Table 21.2.1: Life-of-Mine Operating Cost Summary

Description	US$/t Mill Feed	LoM (US$000's)
Mining (US$/t mined)	1.18	635,356
Mining	11.38	635,356
Processing	14.45	811,997
Tailings	0.47	26,309
Support	5.42	302,724
Total	**$31.83**	**1,776,387**

Source: SRK, 2015

21.2.1 Mine Operating Costs

SRK estimated the mine operating costs on the prepared production schedule and selected mine equipment fleet. Table 21.2.1.1 present the summary of the mine operating costs.

Table 21.2.1.1: Mine Operating Cost Summary

Description	US$/t Mill Feed	LoM (US$000's)
Drilling	0.85	47,547
Blasting	1.79	100,084
Loading	1.31	72,975
Hauling	5.67	316,320
Roads & Dumps	1.37	76,662
Labor	3.61	201,487
Subtotal Open Pit	**$14.61**	**$815,076**
Cost Capitalized to Pre-Stripping	(3.23)	**(179,719)**
Total Open Pit	**$11.38**	**$635,356**

Source: SRK, 2015

The cost of US$1.88/t mined is the result of an assessment of equipment operating hours, estimate of consumables of mine equipment, and mine operations and quantities of labor to manage and execute these operations.

21.2.2 Process Operating Costs

Mineral processing operating costs were prepared by SRK and Tables 21.2.2.1 and 21.2.2.2 presents estimated mineral processing plant and tailings storage facility operating costs.

Table 21.2.2.1: Process Plant Operating Costs

Description	US$/t Mill Feed	LoM (US$000's)
Labor	1.50	83,711
Comminution Consumables	1.70	94,872
Reagents	4.50	251,133
Power (@ US$0.20/kWh)	6.00	334,844
Maintenance Supplies	0.50	27,904
Other	0.35	19,533
Total Processing	**$14.55**	**$811,997**

Source: SRK, 2015

Table 21.2.2.2: Tailings Capital Costs

Description	US$/ Mill	LoM (US$000's)
Pumping	0.47	26,309
Total Tailings	**$0.47**	**$26,309**

Source: SRK, 2015

21.2.3 Support Operating Costs

Nordgold provided an annual estimate of US$24.75 million for General Facilities costs and Site General and Administrative costs. This cost was estimated at US$5.42/t milled by SRK as shown in Table 21.2.3.1. This unit rate is a placeholder based on similar analogous projects and not a build up from first principles.

Table 21.2.3.1: Support Operating Costs

Description	US$/t Mill Feed	LoM (US$000's)
General Facilities	4.30	240,224
Site G&A	1.12	62,500
Total Support	**$5.42**	**$302,724**

Source: SRK, 2015

22 Economic Analysis

The indicative economic results summarized in this section are based upon work performed by SRK or received from Nordgold in 2015. They have been prepared on an annual basis, 100% equity basis, and are in 2015 U.S. constant dollars.

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

22.1 Principal Assumptions and Input Parameters

A TEM was prepared on an after-tax basis, the results of which are presented in this section. Key criteria used in the analysis are discussed in detail throughout this report. Principal assumptions used are shown summarized in Table 22.1.1.

Table 22.1.1: Basic Model Parameters

Description	Technical Input
Pre-Production Period	2 years
Open Pit Mine Life	13 years
Mine Operating Days per Year	365 days per year
Mill Operating Days per Year	365 days per year
Designed Production Rate	12,500 t/d
Discount Rate	EOP @ 8%
Construction Start Year	2018
Commercial Production Year	2020

Source: SRK, 2015

The TEM has 2018 as the construction start year which reflects an assumption that a positive result from a FS and an investment decision to proceed with the Project will be made by the end of 2017. All costs incurred to that point are considered sunk with respect to this analysis.

22.2 Cashflow Forecasts and Annual Production Forecasts

The following tables contain the production and cost information developed for the Project. Table 22.2.1 is a summary of the estimated mine production over a 13-year mine life.

Table 22.2.1: Life-of-Mine Production Summary

Description	Value	Units
Mine Production		
Mill Feed Mined	55,807	kt
Waste Mined	281,498	kt
Total Material Mined	337,306	kt
Strip Ratio	5.0	w:o
Daily Mining Capacity	77,010	t/d
RoM Grade	1.80	g/t
Contained Gold	3,234	koz

Source: SRK, 2015

A summary of the estimated process plant production for the Project is contained in Table 22.2.2.

Table 22.2.2: Life-of-Mine Process Production Summary

Description	Value	Units
Mill Production		
Total Mill Feed Processed	55,807	kt
Daily Process Capacity	12,534	t/d
Processed Grade	1.80	g/t
Contained Gold	3,234	koz
Recovery	94.9%	%
Recovered Gold	3,069	koz

Source: SRK, 2015

The PEA results, shown in Table 22.2.3, indicate an after-tax NPV 8% of US$324 million and IRR of 23.0% with total All-In Sustaining Cost (AISC) of US$711/oz. Initial capital is estimated at US$366 million, sustaining capital at US$216 million (capitalized stripping, mine maintenance, and TSF raise costs), and a closure/reclamation capital cost estimated at US$25 million. The following provides the basis of the SRK LoM plan and preliminary economics:

- A mine life of 13 years;
- A constant LoM gold market price of US$1,200/oz which is the April 2015 monthly average spot close price;
- Doré refining/selling assumptions:
  - 99.5% payable;
  - US$1/oz selling/refining plus transportation/insurance costs;
- Royalties/taxation inputs per this section;
- Capital and operating costs described in Section 21;
- Working capital assumptions:
  - 7 days accounts receivable (A/R);
  - 30 days accounts payable (A/P); and
  - 60 days consumable inventory.

Preliminary economic results and estimated AISC calculations are summarized in Tables 22.2.3 and 22.2.4 and a full LoM annual cash flow forecast is presented in Table 22.2.5.

Table 22.2.3: Life-of-Mine After-Tax Indicative Economic Results (in US$000's)

Description	Value
Market Prices	
Gold (US$/oz)	$1,200
Revenue	
Payable Gold (koz)	3,054
Total Revenue	**$3,664,612**
Operating Costs	
Mining	(635,356)
Processing	(811,997)
Tailings	(26,309)
General Facilities	(240,224)
Site G&A	(62,500)
Selling/Refining	(3,069)
Royalties	(176,082)
Total Operating Costs	**($1,955,538)**
Operating Margin (EBITDA)	**$1,709,074**
Taxes	
Income Tax	(345,397)
Total Taxes	**($345,397)**
Working Capital	(0)
Operating Cash Flow	**$1,363,677**
Capital	
Initial Capital	(366,425)
Sustaining Capital	(216,102)
Reclamation/Salvage Capital	(25,000)
Total Capital	**($607,527)**
Metrics	
Free Cash Flow	$756,150
NPV @: 8%	$324,430
IRR	23.0%
Undiscounted Payback from Start of Comm. Prod. (Years)	3.6
AISC ($/oz)	$711

Source: SRK, 2015

Table 22.2.4: Life-of-Mine All-In Sustaining Cost (AISC) Contribution

Description	US$/oz
Production Cash Costs	562
Off-Site G&A Costs	20
3rd Party Royalties	30
Gov't Production Tax	28
Subtotal Cash Operating Costs	**$640**
Capitalized Mine Maintenance/TSF Raises	21
Capitalized Exploration	0
Capitalized Stripping	50
Subtotal Sustaining Capital	**$71**
Total AISC	**$711**

Source: SRK, 2015

Table 22.2.5: Life-of-Mine Annual Cash Flow Forecast

Economic Model Summary

srk consulting

Company: **Nordgold**
Business Unit: **Montagne d'Or**
Analysis Type: **PEA**

YEAR				2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Project Timeline			LoM	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Production Timeline			Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Discount Factors	EOP @ 8%	US$ & Metric Units	or Avg.	1.0000	0.9259	0.8573	0.7938	0.7350	0.6806	0.6302	0.5835	0.5403	0.5002	0.4632	0.4289	0.3971	0.3677	0.3405	0.3152	0.2919	0.2703
Market Prices																					
Gold		US$/oz	**$1,200**	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
Physicals																					
Total Mill Feed Mined		kt	55,807	-	450	5,500	5,500	5,500	5,276	4,500	4,500	6,079	5,739	5,186	5,500	2,078	-	-	-	-	-
Total Waste Mined		kt	281,498	-	10,550	25,000	24,504	24,982	35,000	35,000	31,799	26,438	20,209	23,934	20,523	3,560	-	-	-	-	-
Total Material Mined		kt	337,306	-	11,000	30,500	30,004	30,482	40,276	39,500	36,299	32,517	25,948	29,120	26,023	5,637	-	-	-	-	-
Strip Ratio		w/o	5.0	-	23.4	4.5	4.5	4.5	6.6	7.8	7.1	4.3	3.5	4.6	3.7	1.7	-	-	-	-	-
Total Mill Feed Tonnes, Processed		kt	55,807	-	-	4,575	4,563	4,563	4,563	4,575	4,563	4,563	4,563	4,575	4,563	4,563	4,563	1,020	-	-	-
Gold Grade, Processed		g/t	1.80	-	-	1.99	2.03	1.97	2.02	2.02	2.07	1.66	1.86	2.11	1.94	1.33	0.84	0.84	-	-	-
Contained Gold, Processed		koz	3,234	-	-	293	298	290	296	297	303	244	272	310	285	195	124	28	-	-	-
Average Recovery, Gold		%	94.9%	--	--	94.8%	94.8%	94.9%	94.9%	94.9%	94.9%	94.9%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	--	--	--
Recovered Gold, Dore		koz	3,069	-	-	277	282	275	281	282	288	231	259	294	271	186	118	26	-	-	-
Payable Gold, Dore		koz	3,054	-	-	276	281	273	279	281	287	230	257	293	269	185	117	26	-	-	-
Cash Flow																					
Total Revenue		**$000s**	**3,664,612**	**-**	**-**	**331,097**	**337,108**	**327,966**	**335,039**	**336,665**	**343,899**	**276,321**	**308,686**	**351,582**	**322,980**	**221,615**	**140,296**	**31,360**	**-**	**-**	**-**
Mining Cost		$000s	(635,356)	-	-	(52,297)	(54,601)	(55,740)	(71,295)	(69,881)	(67,336)	(64,004)	(58,090)	(61,272)	(52,947)	(20,669)	(7,224)	-	-	-	-
Process Cost		$000s	(811,997)	-	-	(66,566)	(66,384)	(66,384)	(66,384)	(66,566)	(66,384)	(66,384)	(66,384)	(66,566)	(66,384)	(66,384)	(66,384)	(14,839)	-	-	-
Tailings Cost		$000s	(26,309)	-	-	(212)	(2,593)	(2,593)	(2,593)	(2,487)	(2,121)	(2,121)	(2,866)	(2,705)	(2,445)	(2,593)	(979)	-	-	-	-
General Facilities Cost		$000s	(240,224)	-	-	(19,791)	(19,718)	(19,725)	(19,719)	(19,787)	(19,712)	(19,769)	(19,741)	(19,774)	(19,729)	(19,814)	(19,882)	(3,062)	-	-	-
Site G&A Cost		$000s	(62,500)	-	-	(5,000)	(5,000)	(5,000)	(5,000)	(5,000)	(5,000)	(5,000)	(5,000)	(5,000)	(5,000)	(5,000)	(5,000)	(2,500)	-	-	-
Refining/Selling Cost		$000s	(3,069)	-	-	(277)	(282)	(275)	(281)	(282)	(288)	(231)	(259)	(294)	(271)	(186)	(118)	(26)	-	-	-
Direct Cash Costs		**$000s**	**(1,779,456)**	**-**	**-**	**(144,143)**	**(148,578)**	**(149,717)**	**(165,272)**	**(164,003)**	**(160,842)**	**(157,510)**	**(152,340)**	**(155,612)**	**(146,777)**	**(114,647)**	**(99,588)**	**(20,427)**	**-**	**-**	**-**
Royalties		$000s	(176,082)	-	-	(17,027)	(17,337)	(16,866)	(17,230)	(17,314)	(17,686)	(14,210)	(13,099)	(14,919)	(13,706)	(9,404)	(5,953)	(1,331)	-	-	-
Indirect Cash Costs		**$000s**	**(176,082)**	**-**	**-**	**(17,027)**	**(17,337)**	**(16,866)**	**(17,230)**	**(17,314)**	**(17,686)**	**(14,210)**	**(13,099)**	**(14,919)**	**(13,706)**	**(9,404)**	**(5,953)**	**(1,331)**	**-**	**-**	**-**
Total Operating Expense		**$000s**	**(1,955,538)**	**-**	**-**	**(161,171)**	**(165,915)**	**(166,583)**	**(182,502)**	**(181,317)**	**(178,528)**	**(171,721)**	**(165,439)**	**(170,531)**	**(160,482)**	**(124,051)**	**(105,541)**	**(21,758)**	**-**	**-**	**-**
Operating Margin		**$000s**	**1,709,074**	**-**	**-**	**169,926**	**171,193**	**161,383**	**152,537**	**155,348**	**165,371**	**104,600**	**143,247**	**181,051**	**162,498**	**97,564**	**34,754**	**9,602**	**-**	**-**	**-**
Earnings Before Interest, Taxes & Depreciation		**$000s**	**1,709,074**	**-**	**-**	**169,926**	**171,193**	**161,383**	**152,537**	**155,348**	**165,371**	**104,600**	**143,247**	**181,051**	**162,498**	**97,564**	**34,754**	**9,602**	**-**	**-**	**-**
Depreciation Allowance		$000s	(582,527)	-	-	(38,173)	(40,922)	(43,536)	(45,520)	(49,082)	(50,743)	(52,348)	(54,627)	(56,225)	(57,498)	(19,906)	(17,331)	(56,617)	-	-	-
Other Non-Cash Tax Adjustments		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Earnings Before Taxes		**$000s**	**1,126,547**	**-**	**-**	**131,753**	**130,271**	**117,847**	**107,017**	**106,266**	**114,628**	**52,253**	**88,620**	**124,826**	**105,000**	**77,658**	**17,424**	**(47,015)**	**-**	**-**	**-**
Income Tax		$000s	(345,397)	-	-	(38,291)	(37,876)	(34,397)	(31,365)	(31,155)	(33,496)	(16,031)	(26,214)	(36,351)	(30,800)	(23,144)	(6,279)	-	-	-	-
Net Income		**$000s**	**781,150**	**-**	**-**	**93,462**	**92,395**	**83,450**	**75,652**	**75,112**	**81,132**	**36,222**	**62,407**	**88,474**	**74,200**	**54,513**	**11,145**	**(47,015)**	**-**	**-**	**-**
Non-Cash Add Back - Depreciation		$000s	582,527	-	-	38,173	40,922	43,536	45,520	49,082	50,743	52,348	54,627	56,225	57,498	19,906	17,331	56,617	-	-	-
Other Non-Cash Tax Adjustments		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Working Capital		$000s	0	-	-	(12,057)	(379)	110	(1,031)	50	35	1,490	(324)	(1,004)	1,050	3,796	2,429	4,557	1,278	-	-
Operating Cash Flow		**$000s**	**1,363,677**	**-**	**-**	**119,578**	**132,939**	**127,096**	**120,141**	**124,243**	**131,910**	**90,059**	**116,709**	**143,695**	**132,748**	**78,215**	**30,904**	**14,160**	**1,278**	**-**	**-**
Initial Capital		$000s	(366,425)	(130,929)	(235,496)																
Sustaining Capital		$000s	(216,102)			(15,304)	(27,492)	(26,136)	(19,840)	(35,623)	(16,609)	(16,046)	(22,792)	(15,986)	(12,725)	(5,812)	(1,736)	-	-	-	-
Closure/Reclamation/Salvage Capital		$000s	(25,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(12,500)	(12,500)	-
Total Capital		**$000s**	**(607,527)**	**(130,929)**	**(235,496)**	**(15,304)**	**(27,492)**	**(26,136)**	**(19,840)**	**(35,623)**	**(16,609)**	**(16,046)**	**(22,792)**	**(15,986)**	**(12,725)**	**(5,812)**	**(1,736)**	**-**	**(12,500)**	**(12,500)**	**-**
Metrics																					
Economic Metrics																					
a) Pre-Tax																					
Free Cash Flow		$000s	**1,101,547**	(130,929)	(235,496)	142,565	143,323	135,357	131,666	119,775	148,797	90,044	120,131	164,061	150,823	95,547	35,447	14,160	(11,222)	(12,500)	-
Cumulative Free Cash Flow		$000s		(130,929)	(366,425)	(223,861)	(80,538)	54,819	186,485	306,260	455,057	545,100	665,231	829,292	980,115	1,075,662	1,111,109	1,125,269	1,114,047	1,101,547	1,101,547
NPV @	8.00%	$000s	**536,452**	(130,929)	(218,052)	122,226	113,774	99,491	89,610	75,478	86,822	48,648	60,095	75,992	64,686	37,943	13,034	4,821	(3,537)	(3,649)	-
Cumulative NPV		$000s		(130,929)	(348,981)	(226,755)	(112,981)	(13,489)	76,120	151,599	238,420	287,068	347,163	423,155	487,841	525,784	538,818	543,638	540,101	536,452	536,452
IRR		%	**31.7%**																		
Undiscounted Payback From Start of Comm. Prod.		Years	**2.6**			-	-	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
PI @	8.00%	NPV / (PW of TC)	**1.09**	130,929	218,052	13,121	21,824	19,211	13,503	22,449	9,691	8,669	11,402	7,405	5,457	2,308	638	-	3,941	3,649	-
b) After-Tax																					
Free Cash Flow		$000s	**756,150**	(130,929)	(235,496)	104,274	105,447	100,960	100,301	88,620	115,301	74,013	93,917	127,710	120,023	72,403	29,168	14,160	(11,222)	(12,500)	-
Cumulative Free Cash Flow		$000s		(130,929)	(366,425)	(262,151)	(156,705)	(55,745)	44,556	133,177	248,478	322,491	416,408	544,117	664,141	736,544	765,712	779,871	768,650	756,150	756,150
NPV @	8.00%	$000s	**324,430**	(130,929)	(218,052)	89,398	83,707	74,208	68,263	55,846	67,277	39,987	46,982	59,154	51,476	28,752	10,725	4,821	(3,537)	(3,649)	-
Cumulative NPV		$000s		(130,929)	(348,981)	(259,583)	(175,876)	(101,668)	(33,404)	22,441	89,719	129,706	176,687	235,842	287,318	316,070	326,795	331,616	328,078	324,430	324,430
IRR		%	**23.0%**																		
Undiscounted Payback from Start of Comm. Prod.		Years	**3.6**			-	-	-	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.6	3.6
PI @	8.00%	NPV / (PW of TC)	**0.66**	130,929	218,052	13,121	21,824	19,211	13,503	22,449	9,691	8,669	11,402	7,405	5,457	2,308	638	-	3,941	3,649	-
Operating Metrics																					
Mine Life		Years	13																		
Average Mining Rate (Mill Feed + Waste)		MTPA	40,276																		
Average Processing Rate		MTPA	4,575																		
Mining Cost		$ / t mined	$1.88	$ -	$ -	$ 1.71	$ 1.82	$ 1.83	$ 1.77	$ 1.77	$ 1.86	$ 1.97	$ 2.24	$ 2.10	$ 2.03	$ 3.67	$ -	$ -	$ -	$ -	$ -
Mining Cost		$ / t mill	$11.38	$ -	$ -	$ 9.51	$ 9.93	$ 10.13	$ 13.51	$ 15.53	$ 14.96	$ 10.53	$ 10.12	$ 11.81	$ 9.63	$ 9.95	$ -	$ -	$ -	$ -	$ -
Processing Cost		$ / t mill	$15.02	$ -	$ -	$ 14.60	$ 15.12	$ 15.12	$ 15.12	$ 15.09	$ 15.01	$ 15.01	$ 15.18	$ 15.14	$ 15.09	$ 15.12	$ 14.76	$ 14.55	$ -	$ -	$ -
Support Cost	21%	$ / t mill	$5.42	$ -	$ -	$ 4.33	$ 4.32	$ 4.32	$ 4.32	$ 4.32	$ 4.32	$ 4.33	$ 4.33	$ 4.32	$ 4.32	$ 4.34	$ 4.36	$ 3.00	$ -	$ -	$ -
Sales Metrics																					
LOM Au Dore Sales		koz	3,054	-	-	276	281	273	279	281	287	230	257	293	269	185	117	26	-	-	-
LOM Avg. Annual Dore Sales		koz / yr	265																		
LOM All-In Sustaining Costs (AISC)		$ / oz	$711																		

Source: SRK, 2015

22.3 Taxes, Royalties and Other Interests

The mining taxation regime in the French Tax Code is under discussion at the time of writing this report. A new mining code could enter into force as early as this year but this date is not certain. Thus, royalties and income taxes have been calculated for the Project with best efforts assumptions provided by Nordgold and SRK.

22.3.1 Royalties

- French Guiana government production taxes which currently net to US$28/oz;
- Euro Ressources' NSR royalty of 1.8% up to 2 Moz, and 0.9% after 2 Moz;
- Sandstorm Resources' 1% NSR royalty; and
- Overall effective NSR royalty rate is estimated 5.2% up to 2 Moz and 4.3% afterwards until the end of production.

22.3.2 Income Taxes

- Assume French Guiana national income tax rate, currently 33.5%, will be reduced to 28% rate in Year 2020;
- Assume 10% straight line depreciation of all capital starting in the first year of commercial production with final write-off of remaining depreciation in the last year of production; and
- Assume no corporate income tax exemption during the first years of production.

22.4 Sensitivity Analysis

Sensitivity analysis for key economic parameters is shown in Tables 22.4.1 and 22.4.2. The Project is nominally most sensitive to market prices (revenues). The Project's sensitivities to capital and operating costs are similar but slightly more susceptible to operating costs. In addition, only a 10% reduction in operating or capital costs or a 5% increase in gold price is enough to bring the Project after-tax IRR to over 25% which is an acceptable target to many mining investors.

Table 22.4.1: Sensitivity Analysis of After-Tax NPV 8% (US$ million)

NPV@8% (US$ Millions)	-20%	-15%	-10%	-5%	Base	5%	10%	15%	20%
Revenue	17	94	171	248	324	401	478	555	632
Operating Costs	471	434	398	361	324	288	251	215	178
Capital Costs	406	385	365	345	324	304	284	264	243

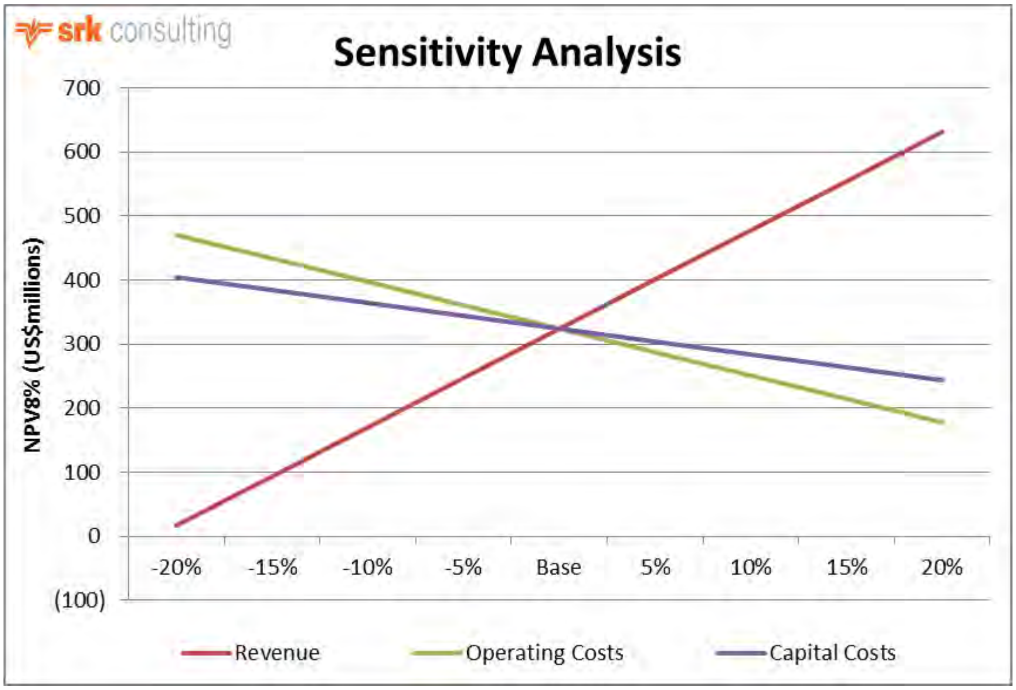
Source: SRK, 2015

Table 22.4.2: Sensitivity Analysis of After-Tax IRR

IRR	-20%	-15%	-10%	-5%	Base	5%	10%	15%	20%
Revenue	8.9	12.9	16.5	19.9	23.0	26.1	29.0	31.8	8.9
Operating Costs	28.7	27.3	25.9	24.5	23.0	21.6	20.0	18.5	28.7
Capital Costs	30.2	28.2	26.3	24.6	23.0	21.6	20.3	19.0	30.2

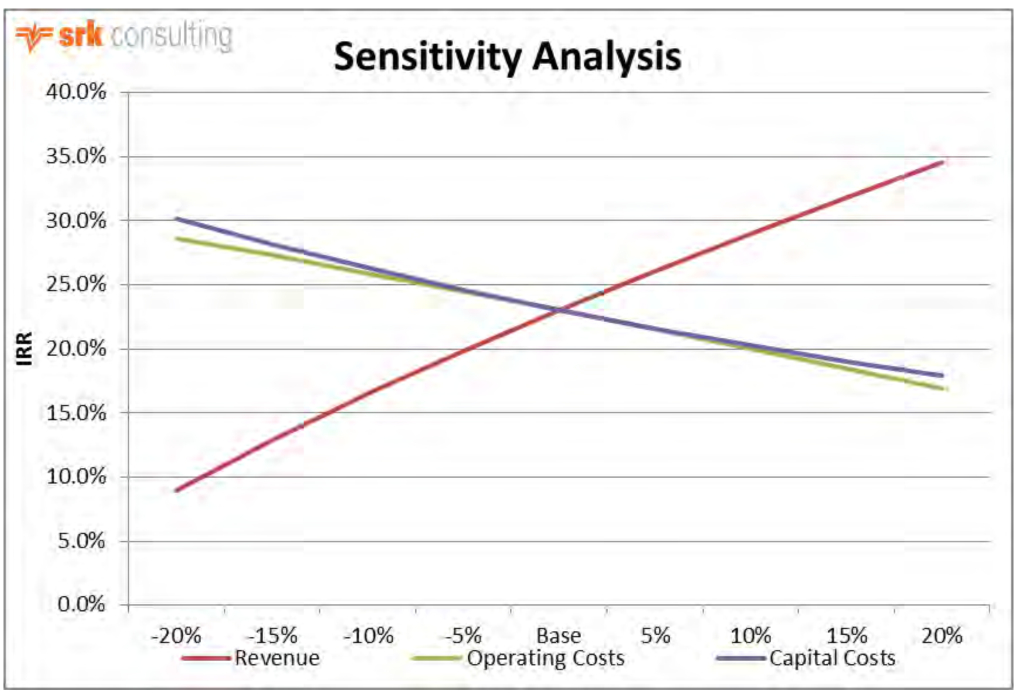
Source: SRK, 2015

Figures 22.4.1 and 22.4.2 also show that only a LoM gold price of US$947/oz would make the Project break even in terms of NPV 8% with operating cost and capital cost held constant.



Source: SRK, 2015

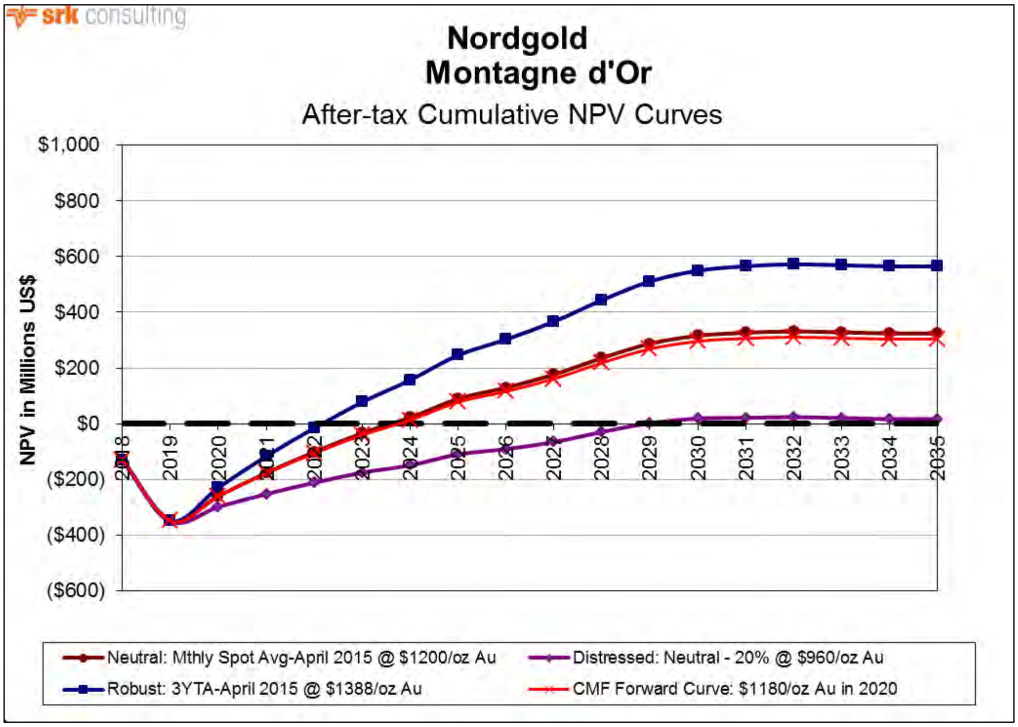
Figure 22.4.1: Project NPV 8% Sensitivities (US$ million)



Source: SRK, 2015

Figure 22.4.2: Project IRR Sensitivities

Additional gold price sensitivity analyses are shown in Figure 22.4.3 with after-tax Project NPV 8% at constant "Robust" prices (April 2015 - 3 year trailing average of US$1,388/oz), and a constant "Distressed" prices (80% of neutral gold price, which is the April 2015 monthly average spot close price, equal to US$960/oz). Furthermore, SRK incorporated a forward price curve sensitivity using Consensus Economics' "Consensus Market Forecast" (CMF Forward Curve), which shows US$1,180/oz in 2020. All told, the after-tax Project NPV 8% changes approximately US$1.1 million for every US$1 change in gold price, either upwards or downwards. In addition, Table 22.4.3 also shows price sensitivity at a series of additional discrete price points.



Source: SRK, 2015

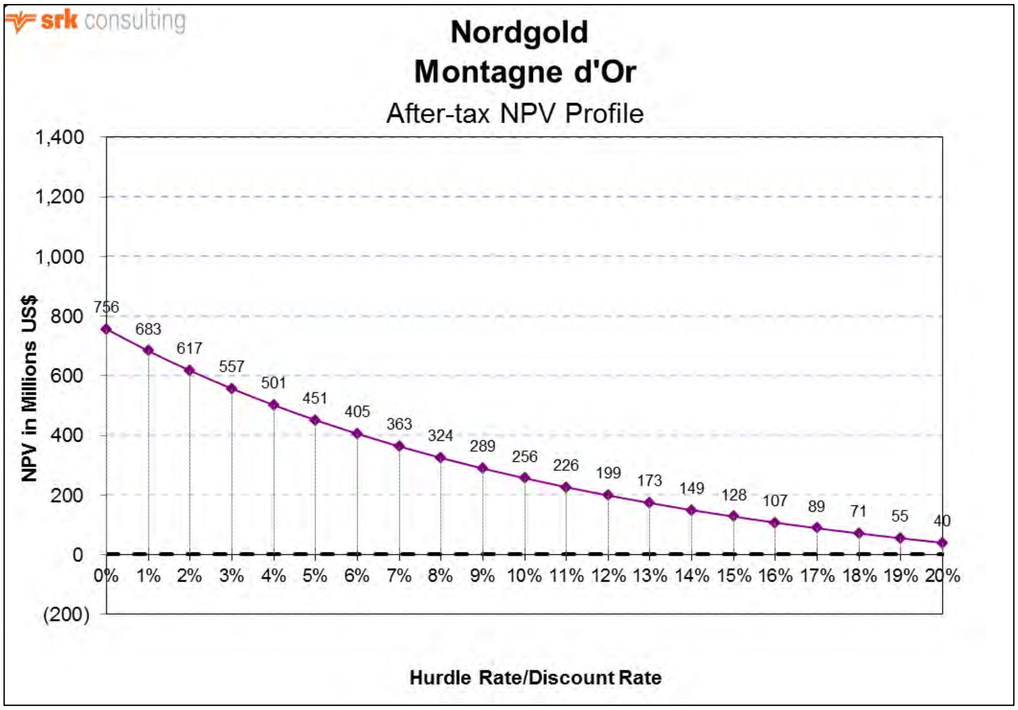
Figure 22.4.3: Project NPV 8% Sensitivities at Varying Gold Prices

Table 22.4.3: Sensitivity Analysis at Various Gold Price Points

Gold Price (US$/oz)	NPV@8% (US$ millions)	IRR (%)
947	$0 (Breakeven)	8.0
1,000	68,495	11.6
1,100	196,471	17.6
1,200	324,430	23.0
1,300	452,388	28.1
1,400	580,347	32.8

Source: SRK, 2015

A sensitivity analysis of discount rates was warranted due to the remote location of the Project. Figure 22.4.4 shows that the Project as currently modeled would still be profitable with respect to after-tax NPV at discount rates of 20% or greater.



Source: SRK, 2015

Figure 22.4.4: Project NPV Sensitivities at Varying Discount Rates

A final sensitivity analysis was carried out by investigating the option of building a power line alongside the access road that would connect to the national power grid. The Base Case had HFO/Palm Oil power generation with a capital cost of US$33 million (before contingency) and an operating cost of US$0.20/kWh. The power line scenario had a capital cost of US$70.2 million (before contingency) and an operating cost of US$0.11/kWh. While the Project NPV 8% increased from US$327 million to US$354 million with the power line scenario, the Project IRR decreased from 23.2% to 22.6%, Nordgold continues to evaluate grid connection options and plans to develop a more accurate capex estimate before making a final decision during the FS process.

23 Adjacent Properties

There are no significant properties adjacent to the Montagne d'Or prospect.

24 Other Relevant Data and Information

24.1 Project Impacts on Economy

Economic benefits achieved during the mine's operational life are estimated to be:

- Government Production Royalties: US$86 million;
- Corporate Income Tax: US$304 million;
- Project operating cost US$2,144 million; and
- Project Investment $458 million.

The operating costs are the biggest total cost for the Project so it is important to understand the "lifeblood" of the Project, which is essentially feed for the process plant, mine equipment and people working at the site. Table 24.1.1 presents the estimates of major mine consumables for the mine and process plant, plus an estimate of the people employed by the company. Key to the success of the Project will be the infrastructure and permanent road that will be used to get goods and services from the St-Laurent and Cayenne ports. The combination of goods and services plus inflow of skilled and semi-skilled workers will have a significant impact to the French Guiana economy and in particular local business. There is little opportunity for material to be transported from neighboring countries and the main commodity to be flown in and out are people and the gold doré produced on site.

Table 24.1.1: Major Commodities and Direct Labor Estimate For the Project Life

Heading	Unit	Year	(1)	1	2	3	4	5	6	7	8	9	10	11	12
		Pre -Prod	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Mining															
Explosives	t	57,154	617	4,235	4,099	4,844	7,370	6,204	7,016	6,073	5,086	5,137	5,277	1,198	
Lube & Oil	kl	7,036	226	575	601	596	760	760	722	696	637	672	556	183	51
Diesel Fuel	kl	142,036	4,374	11,528	12,155	11,963	15,324	15,458	14,522	14,133	13,026	13,723	11,203	3,597	1,031
Power Fuel	kl	58,240	4,480	4,480	4,480	4,480	4,480	4,480	4,480	4,480	4,480	4,480	4,480	4,480	4,480
Total Fuel	kl	200,276	8,854	16,008	16,635	16,443	19,804	19,938	19,002	18,613	17,506	18,203	15,683	8,077	5,511
Process Plant															
Cyanide	t	46,573		3,889	3,879	3,879	3,879	3,889	3,879	3,879	3,879	3,889	3,879	3,879	3,879
Lime	t	10,958		915	913	913	913	915	913	913	913	915	913	913	913
Flocculant	t	1,096		92	91	91	91	92	91	91	91	92	91	91	91
CN Destruct - Sodium metabisufite	t	35,615		2,974	2,966	2,966	2,966	2,974	2,966	2,966	2,966	2,974	2,966	2,966	2,966
CN Destruct - Copper Sulfate	t	3,288		275	274	274	274	275	274	274	274	275	274	274	274
Labor															
Mine and Mine Maintenance			155	351	358	359	431	426	400	385	353	373	315	164	67
Process Plant				89	89	89	89	89	89	89	89	89	89	89	89
General and Administration				100	100	100	100	100	100	100	100	100	100	100	100
Construction			1000												
Total				**540**	**547**	**548**	**610**	**615**	**589**	**574**	**542**	**562**	**504**	**353**	**256**

Source: SRK, 2015

Explosives

Depending on government regulations, it is likely that explosives will be manufactured on site using an emulsion plant set up by a major explosive supplier such as Dyno Nobel or Orica. It may be possible to import pre-packaged explosives, but given the high amounts needed per year (over 7,300 t/y) it is likely the economics would favor locally manufactured explosives given the length of mine operations. Importation of the raw ingredients would be similar to the importation and transportation of plant consumables via port and road.

Diesel

Fuel will be shipped to the ports of French Guiana where it will be stored and dispatched to the Project site via truck. Assuming a truck can hold 30,000 L, at least two fully loaded fuel trucks would be required to deliver their fuel to site every day, 365 days a year for the LoM. Storage will be required at the two major port facilities plus at the mine site.

Lime

Quicklime will be delivered to the site in bulk by pneumatic tanker and stored in the lime silo. It is anticipated that the quicklime will be slaked in a vendor-supplied package accompanying the silo. The slaked lime will be pumped to the grinding circuit and the cyanide destruction circuit in a ring main. A dust collector will minimize dust emissions during silo filling.

Cyanide

Sodium cyanide will be delivered as briquettes in shipping containers containing approximately 1 t of cyanide each. The containers will be emptied into the cyanide mixing tank and combined with water to dissolve the cyanide to a target strength of 20% NaCN. Sodium hydroxide will be added to the mixing tank prior to cyanide addition in order to maintain a solution pH of 11 to prevent HCN generation. The mixed cyanide solution will be transferred to the storage tank for dosing to the process. Empty cyanide containers will be returned to the vendor.

Grinding Media

Grinding balls will be delivered to site in bulk or 200 L steel drums.

Flocculant

Flocculant for use in the pre-leach and cyanide recovery thickeners will be delivered to site in 25 kg bags. Flocculant will be added to the flocculant plant storage hopper manually. The vendor supplied flocculant mixing plant will automatically mix batches of flocculant and transfer the mixed flocculant to the aging tank after each mixing cycle is complete. Flocculant will be distributed to the thickeners using positive displacement dosing pumps.

Copper Sulfate

Copper sulfate will be delivered in 1 t bulk bags and will be added to the mixing tank using an electric hoist and bag breaker. Fresh water will be added to the mixing tank to dilute the copper sulfate. The solution will be metered to the cyanide destruction and flotation circuits directly from the mixing tank.

Sodium Metabisulfite

Sodium metabisulfite will be delivered in 1 t bulk bags and will be added to the mixing tank using an electric hoist and bag breaker. An air exhaust fan will draw dust and fumes away from this area as

SO_2 gas is evolved and the dust can cause skin irritation. Fresh water will be used to mix the sodium metabisulfite. The solution will be pumped from the mixing tank to the storage tank for metering to the cyanide destruction circuit by dosing pump.

Food, Supplies, Leisure, Transportation, Business Services

Even through the Project is in a remote location the site must be supported by all the usual businesses that revolve around primary industry. With the consumables, people and business flowing through the major centers of French Guiana, the multiplier effect service industry will be significant.

24.2 Employment Income Sources

The development of the Montagne d’Or is expected to provide significant new employment and business benefits, during its expected 13 year mine life that is likely to extend even longer.

It is estimated that the Montagne d’Or will employ up to 600 people depending on the mine phase: construction, operations, or reclamation. During construction, it is likely up to 1,000+ workers including contractors will be required. During operations, average personnel requirements are estimated at 500 people per year with 350 people on site at any one time. It is estimated that up to 90% of the workforce during the operations phase could come from French Guiana and/or France.

Types of employment include: mine workers, management and technical personnel, general and administrative personnel, support staff (health and safety, environmental, warehousing, camp workers), process operating plant staff, laboratory and maintenance personnel. It is anticipated that the mine will operate 365 days per year. Shut-downs will occur from time to time for maintenance.

Nordgold will attempt to put in place a local hiring strategy that preferentially hires French Guiana residents (with the same levels of education, skill, aptitude and experience) over non-residents (people who do not live in the French Guiana). This strategy is intended to benefit French Guiana residents financially through stable employment while also reducing potential effects on housing, families and public services that would arise from an influx of new residents to the region.

SRK estimated US$202 million in LoM mine wages, US$83 million in LoM plant wages, and then US$25 million per year in G&A costs that may assume 30% as labor adding another US$100 million over the LoM, so roughly US$400 million in wages are to be paid to the Montagne d’Or labor through the LoM.

These values do not include the multiplier effect of downstream contractor and vendor support required to supply the mine with goods and services needed for operations.

24.3 Public Health and Safety

Employee safety and a healthy work environment are paramount concerns for Nordgold and the company is committed to maintaining a safe, healthy and industrious workplace by developing safe work procedures and policies for employees to follow.

Safety training is required for all employees and contractors. Nordgold will distribute training programs to the workforce based on position requirements and is still a workplace culture of benefit to primary industry in French Guiana.

24.4 Long-Term Dependence and Sustainability

The Montagne d'Or expected mine life is expected to be 13+ years. The lifespan is variable dependent mainly upon the establishment of more resources with subsequent reserves through exploration, an increase or decrease in the price of gold, increase or decrease of costs, and the production rate.

By operating a mine, the development of the mining industry in French Guiana can begin. Implementing this kind of mining operation could give impetus to the industry as a whole and reinforce skills which would be of service to other mining operators in different parts of the country;

By having a positive cashflow operation of the scale proposed by the mine, the cashflows will help to mitigate the increasing imbalance between revenue and government expenses in the region;

Mining operations will require skilled and unskilled labor, so the mine will help to train qualified workers in various technical and professional sectors. The skilled labour pool is currently very limited in French Guiana.

24.5 Cultural Property

There are several issues that are local to French Guiana that will be moderated by the establishment of a modern mining operation:

- Helping to fight against illegal gold mining. There would be numerous advantages to eliminating this clandestine activity from the Project area;
- Reduce gold pillaging in French Guiana as well as its illegal trade from Brazilian miners;
- Prevent mercury from being discharged into the environment, generated by its use in gold extraction by illegal miners (its use has been banned since 2006);
- Stop damage to the Lucifer Dékou-Dékou Integral Biological Reserve, to the bottom of valleys and to creeks, as well as wildlife poaching by illegal miners;
- Restoring the degraded sites in the Lucifer Dékou-Dékou Integral Biological Reserve and along the creeks: stopping illegal gold mining will open passive and active restoration opportunities for disturbed sites both inside and outside the reserve;
- Control access to the Paul-Isnard track from the Apatou crossroad to make access more difficult for illegal gold miners and to limit natural resource harvesting pressure (hunting, fishing, gathering, etc.); and
- Provide detailed flora and fauna surveys in advance of any mining activity. This provides very detailed information about wildlife that would otherwise remain unstudied.

25 Interpretation and Conclusions

25.1 Geology and Resources

Geology and resources interpretations and conclusions are:

- Columbus has completed an industry standard exploration drilling program over an area of approximately 1 1/4 km^2;
- The results of the drilling have supported an industry standard resource estimation; and
- Whittle™ pit shell optimizations host an Indicated Mineral Resource of 83 Mt at an average Au grade of 1.455 g/t containing 3.9 Moz of gold and an additional Inferred Mineral Resource of 22 Mt at an average Au grade of 1.550 g/t containing 1.1 Moz of gold.

25.2 Mining

Mining interpretations and conclusions are:

- The open pit mining operation envisaged for Montagne d'Or will be comprised of traditional open pit mining equipment utilizing appropriately sized loaders and two sizes of mining trucks. The operation is moderately sized to produce 12,500 t/d of mill feed with a low grade stockpile to ensure high grade mill feed is processed first. The mine plan utilizes a phased bench sequence approach that follows precedence relationships, maintains a reasonable balanced fleet, provides approximately 300 koz of gold per year at a mining cost of US$2.37/t or US$815 million for the 11 years of full mine production. To achieve this mine capital is estimated at US$86 million over the LoM and US$54 million initially.
- The bulk properties for mill feed, waste and saprolite have been considered in the selection of a dual fleet comprising of 100 t class trucks paired with 40 t articulated dump trucks for pioneering, saprolite and selective mining. The two fleets provide a compromise in reducing labor costs using bulk earthworks, but also having the increased capacity to deal with high rainfall, poor traction, steep terrain and the possibility for use of selective mining.
- The mine production rate targets a 10 year plus mine life given the known resources that are likely to grow in the future. To achieve this, the mine is expected to excavate 80,000 t/d in order to meet the mill requirements. To meet this target roads systems, phase sequencing and continuous operations will be required to achieve required production levels. Optimization of haul cycle times will be key in reducing the mining costs.

25.3 Metallurgy and Processing

Metallurgy and processing interpretations and conclusions are:

- The metallurgical test program was conducted on two master composites formulated from available whole core intervals representing the UFZ and the LFZ, as well as selected variability composites.
- Three process options, including whole-ore cyanidation, a combination of gravity concentration followed by cyanidation of gravity tailing, and gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate, were investigated on two master composites, and the preferred process option and optimal conditions were further verified on ten variability test composites.

- Processing by gravity concentration followed by cyanidation of the gravity tailings yielded the highest overall gold recoveries and was selected at the preferred process option. Gold recovery is projected at about 95% with this process option.

25.3.1 Recovery Methods

Recovery interpretations and conclusions are:

- The selected process flowsheet will include gravity concentration followed by cyanidation of the gravity tailings to recover the contained gold and silver values, and will incorporate process unit operations that are standard to the industry, including: crushing, grinding, agitated cyanide leaching, gold and silver adsorption onto activated carbon, gold and silver desorption, electrowinning and refining.
- Process operating costs are estimated at US$14.55/t processed. Operating costs have been estimated by major category (labor, power, consumables, etc.) and are based on a throughput capacity of 12,500 t/d. The major contributors to operating cost are power and reagents.
- The capital cost for the 12,500 t/d process plant is estimated at US$136.7 million and is considered at a conceptual level with a +/-50% level of accuracy.

25.4 Environmental and Social

Environmental and social interpretations and conclusions are:

- Illegal artisanal mining in the area continues to degrade the surface water resources in the region, including mercuric contamination, and increased erosion and sedimentation. Stakeholder sentiment is that a large-scale, authorized mining operation in the region will bring much needed economic benefits, but also discourage and drive off the illegal miners.
- The operation is currently permitted for all of the activities associated with the exploration program from which this PEA has been prepared. Additional permitting will be necessary in order to move into the exploitation phase of the Project. Initiation of this permitting will likely occur during the preparation of FS, and will include a detailed ESIA based on the FS design of the operation.
- In addition to the land restrictions presented by the SDOM, the Project is located adjacent to a nature reserve, the Réserve Biologique Domaniale Lucifer Dékou-Dékou, managed by the ONF. Its Management Plan from the ONF is yet to be ratified, so there is little guidance or decisions regarding the use of land and allowable activities within the reserve. The boundaries of this reserve overlap four of the eight Paul Isnard mineral concessions however only one of these concessions is important to the project. Since these concessions already exist, and there has been continued exploration and mining activity in the area for over 100 years, the ONF has agreed to create several zones within the reserve boundaries where mining is permitted. The Montagne d'Or deposit itself is within a zone where open pit mining is permitted and the outer limit of the resource pit shell is located approximately 240 m from the reserve boundary.
- Through its association as a Department of France, the Project will be subject to various European Union directives and guidelines.

- The *Preliminary Environmental Report* (WSP, 2015) provides an overview of the environmental and socio-economic issues for the Project, as well as a preliminary indication of the positive and potentially negative impacts associated with the planned operation. The report is intended to provide direction for the continuing environmental assessment process, and guide the environmental authorities with the information required to determine the range of information and degree of detail needed in the formal impact assessment.
- Based on the preliminary geochemical characterization program initiated for mineralized material, waste rock and tailings, the potential for acid generation and leaching of metals remains a concern at this stage of the project, and will need to be considered during design and development of the mine with respect to appropriate waste rock and tailings management.
- By law, reclamation of the mine site following closure is required. The operator is required to restore the site to a state that is, at a minimum, similar to that described in the Baseline Report. Given the current lack of mine design information, the costs associated with closure of the Montagne d'Or Project have been estimated at approximately US$25 million based on similar nature and extent of the operations to projects previously evaluated by SRK. This number will be refined using actual mine designs and country-specific costing rates during development of the project FS.
- Even though French Guiana (through its connection with France) is a Designated Country with respect to the Equator Principles, Nordgold has committed to ensuring that Montagne d'Or is in compliance with international standards and guidelines, to the extent practicable, given the potential for international investment in the Project.

25.5 Projected Economic Outcomes

- The Project estimates economic results using US$1,200/oz gold price with NPV 8% at US$324 million and 23.0% IRR. The Project, as currently designed with an Initial Capital cost of US$366 million for the 13 year mine life year mine life at a total cash cost of US$711/oz;
- For the first 11 years when stockpiles are not fed to the mill, the annual recovered gold ounce production is approximately 265 koz/yr;
- The Project NPV 8% changes by approximately US$1.1 million per dollar change in gold price; and
- Mining taxation assumptions should be investigated further due to current uncertainty in French tax code.

25.6 Foreseeable Impacts of Risks

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Due to the nature of the PEA, SRK has used its experience to reasonably estimate capital and operating costs for the determination of project economics. These estimates are not supported by detailed quotes or engineering studies to reduce the accuracy from +- 40% as defined in this PEA.

As the detailed fieldwork and process engineering studies commence the accuracy of these costs are likely to improve thus reducing the financial project risk.

Many of the risk items such as water balance, geotechnical stability of saprolite, source of road base material, infrastructure design, process design, commodity price, import duties, labor rates, NAG/PAG management (acid generation management), hydrological modelling, water treatment, sterilization, site layout, explosive importation, government policy, community and social issues, NGO's, grade variation, dilution, selective mining units and other project risks are all to be addressed as part of a detailed FS.

The project is burdened by its remote location and lack of infrastructure in a country that is not accustomed to mining. The grade of 2 g/t is reasonable but not considered high, so pushing tonnage through the process plant will be a key aspect. If there is a drop in gold price from US$1,200/oz. Au then the project economics will be stressed if no tax credits from the French government, or other operational efficiencies, can be achieved. The social and environmental factors relating to the project from a non-technical perspective also pose a significant risk to the success of the project.

26 Recommendations

26.1 Recommended Work Programs and Costs

Apart from the exploration and ongoing drilling, the recommendations made in these sections are covered cost-wise as part of a FS program.

26.1.1 Exploration Drilling

A multitask exploration drilling program is proposed. The program will target infill drilling in the areas of the proposed starter pit, infill drilling in the saprolite material and condemnation drilling in the potential areas of infrastructure.

The infill drilling program would be on a 25 m x 50 m grid spacing in the proposed area of the current resource starter pit. The drillholes are proposed to range from 35 to 320 m in length. Many of the holes would be drilled by RC to the maximum depth achievable, and then taken to final depth with core. A total of 17,750 m in 123 drillholes would be required.

The condemnation drilling program will cover three areas of infrastructure including, the proposed plant site, the proposed waste rock site and the proposed tailings facility. The condemnation drilling would be on a 55 m grid pattern and would consist of 75 m long inclined holes at -55° to the north or northeast. A total of 4,900 m in 65 drillholes would be required.

26.1.2 Open Pit Geotechnical Program

The following is a list of geotechnical data and information gaps that should be addressed as a part of advancing the Project to a feasibility-level study:

- Rock strength testing. A rock strength testing program should be; conducted to test each rock type for uniaxial compression, triaxial compression, elastic moduli, and direct shears on fractures.
- Saprolite characterization and testing. This should be done by a soils drilling rig, and collection of "undisturbed" samples using a split spoon sampler. Samples will be submitted for engineering soils classification, moisture, density and residual strength testing;
- Geotechnical specific drillholes that target pit walls at approximately 90º and provide an unbiased orientation to better understand discontinuity sets in the diabase and volcanic rocks;
- Geotechnical model. All available geotechnical data should be domained by geology and fault blocks. The domains should be analyzed and geotechnical parameter distributions should be incorporated into a rock mass model;
- Rock mass strengths should be developed from the testing data and rock mass model;
- Review of major structure intercepts with the mine design. A stability analysis should be conducted with respect to major structures and fault blocks;
- Bench, inter-ramp and overall slope stability analysis should be completed for the open pit design, analyzing each wall orientation and rock mass domain to optimize pit slope angles
- The stability analysis of the pit should incorporate geohydrology and groundwater surface information; and
- All available data and analysis should be documented in a technical report.

26.1.3 Mine Recommendations

Due to the amount of pioneering work that will be conducted in a rainforest environment with considerable amounts of saprolite on the side of a hill, the ability for trucks to operate efficiently will be vital for the successful execution of the mine plan. It is recommended that during the FS, the geomechanical properties of benign rocks (no sulfidation) be tested for suitability as a road course material for haul roads. In addition to the use of some waste rock from the pit, SRK recommends that a quarry site be searched for, either as part of the sterilization drill program or geological interpretation. If a quarry is not possible then a source of laterite that can be screened for fines would also be suitable.

The major revision to the mine plan during a FS would be to spend considerable time on the pioneering road earthworks and pit phase designs to ensure that the trucks always have access to a haulage ramp. Preferably, the ramps are internal to the phase designs, but external pioneering roads are also required. This is a time consuming process outside of the current PEA scope.

The phase design must also be coordinated with the construction of the tailings dam, since the majority of the dam wall will constitute a waste dump for the mining operation. Therefore, the waste material will need to be segregated into NAG and PAG for appropriate disposal.

The open pit is highly sensitive to the location of the southern toe line defining mill feed at depth. The definition of this toe line should be supported by additional drilling to ensure that the pit bottom is supported by actual drill data rather than interpolated block model data. A fence drilling operation should be considered given the sensitivity of the wall and associated stripping penalty for any mistakes.

The PEA is based on the current exploration holes that are generally spaced on a 50 m by 50 m pattern. Because the project has not been mined and the drilling is widely spaced, the true variability of the deposit remains uncertain. SRK recommends In-fill drilling on the western pit extents, which should help determine the variability of the deposit from a grade control perspective, with the models to be changed accordingly.

SRK recommends that a FOS analysis on the pit walls should be done as soon as possible. This will help determine the groundwater and geomechanical properties to be collected that will assist in the generation of final pit wall angles for the FS.

SRK recommends that a NAG/PAG ARD model be built for the classification of the waste rock types that require encapsulated disposal or which can be used for anything else. Metals based accounting should also be considered as part of this exercise as it is evident that there is acid neutralizing potential in some of the waste rocks.

26.1.4 Tailings and Infrastructure

A site water balance should be conducted with the aim of the TSF design to provide a net neutral water balance. This would prevent any discharge that would mean a water treatment plant would not be needed. SRK recommends that further work in this regard be continued in the FS.

The ground conditions of the TSF earthen embankment will require geotechnical investigation for stability purposes. As part of the geotechnical program, some of the soils and earth in the tailings valley should be tested for any potential contamination from artisanal miners as it may need to be stored separately when the tailings area is cleaned.

- Additional geochemical testing on a sample representative of the supernatant pond waters produced when liberated waters from the CIL tailings;
- Additional geochemical testing on samples representative of the waste rock being used for construction of the TSF dam.
- Final design-level subsurface site investigations in select areas including geotechnical laboratory testing, including a study to estimate available borrow material quantities;
- Finalization of the embankment sections including foundation excavation limits for use in final designs;
- Additional subsurface investigation to support a site-wide hydrogeologic evaluation of the Project area;
- Final design-level study and design of the run-off collection channels and ponds;
- Final design of tailings distribution system and water reclaim system considering a potential economic trade-off study for different system options. This will include the preparation of an operations manual for the operation of both of these systems;
- Conduct a dam break analysis in potentially impacted drainages;
- Conduct a site specific seismic hazard assessment;
- Preparation of technical specifications and Construction Quality Assurance and Quality Control (CQA/QC) plans.
- Preparation of the TSF instrumentation and monitoring program; and
- Preparation of an Emergency Action Plan (EAP).

The final design phase will include preparation of design drawings in sufficient detail for use during construction and an updated quantity and cost estimate to a final design-level. Services during construction include resident engineering, home office support, and CQA/QC services.

26.1.5 Environmental

Given the results of the geochemical characterization to date, the program should be expanded to include additional samples of mineralized material and waste rock from around the deposit, as well as post-process tailings. Early indications are that additional active management of waste rock and tailings may be necessary in the hot and humid climate of French Guiana.

A site-wide, soil mercury contamination program should be considered to more accurately define the nature and extent of pre-mine contamination by illegal artisanal mining operations. Exposure to mercury-contaminated soils by future mine workers could present health and safety concerns. Remediation of soils with elevated mercury concentrations could be required (the U.S. Environmental Protection Agency Soil Screening Level for Hg is 23 mg/kg).

26.1.6 Economics

Mining taxation assumptions should be investigated further due to current uncertainty in French tax code.

26.1.7 Costs

Estimated costs for recommended work programs are summarized in Table 26.1.7.1.

Table 26.1.7.1: Summary of Costs for Recommended Work

Item	Units (m)	US$/Unit	Cost (US$)
Exploration Drilling			
Infill RC Drilling	10,000	55	550,000
Infill Core Drilling	4,800	115	550,000
Condemnation Drilling	4,900	55	270,000
Sampling, Logging, Analysis and Overhead	19,700	80	1,580,000
Subtotals	**19,700**	**$150**	**$2,950,000**
Feasibility Study	**1**		**$5,000,000**

Apart from the drilling program, other recommendations made are covered as part of the FS.
Source: SRK, 2015

27 References

Binns RA, Scott SD (1993). Actively forming polymetallic sulfide deposits associated with felsic volcanic rocks in the eastern Manus back-arc basin, Papua New Guinea. Econ. Geol. 88:2226-2236.

Bureau Veritas Commodities Canada Ltd. (2015). Metallurgical Testing to Recover Gold and Silver on Samples from the Montagne d'Or Gold Project, French Guiana. Prepared for: Nordgold. April 6, 2015.

Coffey (2014). Paul Isnard Project, Prepared for Columbus Gold Corporation, August 4, 2014.

Delor C, Lahondère D, Egal E, Lafon J-M, Cocherie A, Guerrot C, Rossi P, Truffert C, Théveniaut H, Phillips D, Gama de Avelar V (2001). *Transamazonian crustal growth and reworking as revealed by the 1:500,000-scale geological map of French Guiana (2nd edition),* Géologie de la France, 2003 no 2-3-4, pp 5-75.

Franklin JM (1999). *Consulting Report: Paul Isnard property, Montagne D'Or region, French Guiana;* Private internal consultant's report prepared for Guiana Resources, S.A, and Golden Star Resources Ltd.

Franklin JM, Bertoni C, Boudrie M, Bout J-P, Costelloe CJ, Francis-Lillié F, Millo I Sauvage JF (2001). *The Paul Isnard gold-copper occurrence, French Guiana: The first volcanogenic massive sulphide occurrence in the Guiana shield?;* in Sherlock R, Logan MAV, VMS Deposits in Latin America, Geological Association of Canada. Mineral Deposits Division Special Publication, 9, 234 -266.

Guiraud J, Jébrak M, Tremblay A, Université de Québec à Montréal (2014). *Montagne d'Or (Paul Isnard, Guyane), Compte rendu de mission; pour Columbus Gold Corp;* 23 Avril 2014.

Glasby GP, Iizasa K, Yuasa M, Usui A (2000). Submarine hydrothermal mineralization on the Izu-Bonin Arc, south of Japan: an overview. Marine Georesources & Geotechnology 18:141-176.

Gold Fields Ltd (2001). *Data review and field visitation of the Paul Isnard Project Area;* Private company internal report; 2001.

Inspectorate, (2015). Metallurgical Testing to Recover Gold and Silver from the Montagne d'Or Gold Project, French Guiana, March 30, 2015.

Inspectorate, (2015). "Mineralogical Study on the Master Composites," January 2015

Milesi JP, Lerouge C, Delor C, Ledru P, Billa M, Cocherie A, Egal E, Fouillac A-M, Lahondère D, Lassere J-L, Marot A, Martel-Jantin B, Rossi P, Tegyev M, Théveniaut H, Thiéblemont D, Vanderhaeghe O (2003). *Gold deposits (gold-bearing tourmalinites, gold-bearing conglomerates, and mesothermal lodes), markers of the geological evolution of French Guiana: geology, metallogeny and stable-isotope constraints;* Géologie de la France, 2003no 2-3-4, pp 257- 290.

Paulick H, Vanko DA, Yeats CJ (2004). Drill core-based facies reconstruction of a deep-marine felsic volcano hosting an active hydrothermal system (Pual Ridge, Papau New Guinea, ODP Leg 193). J. Volcanol. Geotherm. Res. 130:31-50

Ross, Pierre-Simon (2014). Examination of Drill Cores and Lithogeochemical Data from the Montagne d'Or Deposit, Paul Isnard Project, French Guiana. Internal company report, August 2, 2014, 77p.

RSG Global (Jones C, Sperinck M, Hearne J,) (2004). on behalf of Golden Star Resources Limited; *Paul Isnard, Yaou and Dorlin Projects, French Guiana, Independent resource estimations;* January 2004.

Shaw (2001). Section 8/page 39.

Stryhas B (2012). *Updated NI 43-101 Technical Report Paul Isnard Project French Guiana prepared for Columbus Gold Corporation.*

Suter (1999). Paul Isnard & Eau Blanche Deposits – Summary report on historical and recent exploration: private internal company report prepared by Guayanor Resources S.A., 116 p

Vanderhaeghe O, Ledru P, Thiéblemont D, Egal E, Cocherie A, Tegyey M Milési JP (1998). *Contrasting mechanism of crustal growth: Geodynamic evolution of Paleoproterozoic granite-greenstone belts of French Guiana;* Precambrian Research, 92(2), 165-193.

Voicu G, Bardoux M, Stevenson R (2001). *Lithostratigraphy, geochronology and gold metallogeny in the northern Guiana Shield, South America: a review;* Ore Geology Review, 2001, vol. 18, no.3-4, pp. 211-236.

WSP Canada Inc. (2015). Preliminary Environmental Report.

28 Glossary

The Mineral Resources and Mineral Reserves have been classified according to the "CIM Definition Standards for Mineral Resources and Mineral Reserves" (May 10, 2014). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

28.1 Mineral Resources

A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

28.2 Mineral Reserves

A **Mineral Reserve** is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a Prefasibility Study or Feasibility Study.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

A **Proven Mineral Reserve** is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

28.3 Definition of Terms

The following general mining terms may be used in this report.

Table 28.3.1: Definition of Terms

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste in the mineralized material.
Crushing	Initial process of reducing mineralized material particle size to render it more amenable for further processing.
Cut-off Grade (CoG)	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with mineralized material.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of a mineralized material area or stope.
Gangue	Non-valuable components of the mineralized material.
Grade	The measure of concentration of gold within mineralized rock.
Hangingwall	The overlying side of an mineralized material area or slope.
Haulage	A horizontal underground excavation which is used to transport mined material.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LoM Plans	Life-of-Mine plans.
LRP	Long Range Plan.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the mineralized material is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Pillar	Rock left behind to help support the excavations in an underground mine.
RoM	Run-of-Mine.

Term	Definition
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, mineralized material and waste.
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

28.4 Abbreviations

The following abbreviations may be used in this report.

Table 28.4.1: Abbreviations

Abbreviation	Unit or Term
AA	atomic absorption
Ai	Abrasion index
ANFO	ammonium nitrate fuel oil
Ag	silver
Au	gold
AuEq	gold equivalent grade
Bwi	Bond ball mill work index
°C	degrees Centigrade
CCD	counter-current decantation
CIL	carbon-in-leach
CIP	Carbon-in-pulp
CoG	cut-off grade
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
cfm	cubic feet per minute
CNwad	weak-acid dissociable cyanide
ConfC	confidence code
CRec	core recovery
CSS	closed-side setting
CTW	calculated true width
°	degree (degrees)
dia.	diameter
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
FA	fire assay
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
g	gram

Abbreviation	Unit or Term
gal	gallon
g/L	gram per liter
g-mol	gram-mole
gpm	gallons per minute
g/t	grams per tonne
ha	hectares
HDPE	Height Density Polyethylene
hp	horsepower
HTW	horizontal true width
ICP	induced couple plasma
ID2	inverse-distance squared
ID3	inverse-distance cubed
IFC	International Finance Corporation
ILS	Intermediate Leach Solution
kg	kilograms
km	kilometer
km^2	square kilometer
koz	thousand troy ounce
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LFH	Lower Favorable Zone
LHD	Long-Haul Dump truck
LLDDP	Linear Low Density Polyethylene Plastic
LOI	Loss On Ignition
LoM	Life-of-Mine
m	meter
m^2	square meter
m^3	cubic meter
masl	meters above sea level
MARN	Ministry of the Environment and Natural Resources
MBS	Sodium metabisulfite
MDA	Mine Development Associates
mg/L	milligrams/liter
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
MME	Mine & Mill Engineering
Moz	million troy ounces
Mt	million tonnes
MTW	measured true width
MW	million watts
m.y.	million years
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
OSC	Ontario Securities Commission
oz	troy ounce
%	percent
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	probable maximum flood

Abbreviation	Unit or Term
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron
RC	rotary circulation drilling
RoM	Run-of-Mine
RQD	Rock Quality Description
SAG	Semi-autogenous grinding
SEC	U.S. Securities & Exchange Commission
sec	second
SG	specific gravity
SMC	SAG mill comminution
SPT	standard penetration testing
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
t/h	tonnes per hour
t/d	tonnes per day
t/y	tonnes per year
TSF	tailings storage facility
TSP	total suspended particulates
UFZ	Upper Felsic Zone
µm	micron or microns
V	volts
VFD	variable frequency drive
W	watt
XRD	x-ray diffraction
y	year

Appendices

Appendix A: Certificates of Qualified Persons

SRK Denver
Suite 3000
7175 West Jefferson Avenue
Lakewood, CO 80235

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Bart A. Stryhas PhD, CPG # 11034, do hereby certify that:

1. I am a Principal Resource Geologist of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled “Amended NI 43-101 Technical Report Preliminary Economic Assessment, Montagne d’Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana” with an Effective Date of July 8, 2015, prepared for Nord Gold N.V. with Columbus Gold Corporation (the “Technical Report”).
3. I graduated with a Doctorate degree in Structural Geology from Washington State University in 1988. In addition, I have obtained a Master of Science degree in Structural Geology from the University of Idaho in 1985 and a Bachelor of Arts degree in Geology from the University of Vermont in 1983. I am a current member of the American Institute of Professional Geologists. I have worked as a Geologist for a total of 28 years since my graduation from university. My relevant experience includes minerals exploration, mine geology, project development and resource estimation. I have conducted resource estimations since 1988 and have been involved in technical reports since 2004.
4. 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 a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Montagne d’Or property on April 1 to 3, 2014.
6. I am responsible for the preparation of background, geology and resource estimation Sections 2 to 12, 14 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the technical reports listed below:
    - “NI 43-101 Technical Report on Updated Resources, Montagne d’Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana” with an Effective Date of April 11, 2015, prepared for Nord Gold N.V. with Columbus Gold Corporation;
    - “Updated NI 43-101 Technical Report, Paul Isnard Project, French Guiana” with an Effective Date of February 1, 2012, prepared for Columbus Gold Corporation;
    - “Updated NI 43-101 Technical Report on Resources, Columbus Gold Corporation, Paul Isnard Project, French Guiana” with an Effective Date of January 13, 2011, prepared for Columbus Gold Corporation; and
    - “NI 43-101 Preliminary Assessment, Golden Star Resources Ltd., Paul Isnard Project, French Guiana” with an Effective Date of February 29, 2008, prepared for Golden Star Resources Ltd.;
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 31st Day of July, 2015.

"Signed and Sealed"

Bart A. Stryhas PhD, CPG

SRK Denver
7175 West Jefferson Avenue.
Suite 3000
Lakewood, CO 80235

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF AUTHOR

I, Bret C. Swanson, BEng Mining, MAusIMM, MMSAQP do hereby certify that:

1. I am Principal Mining Engineer of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015, prepared for Nord Gold N.V. with Columbus Gold Corporation (the "Technical Report").
3. I graduated with a degree in Bachelor of Engineering in Mining Engineering from the University of Wollongong in 1995. I am a current member of the Mining & Metallurgical Society of America #01418QP. I have worked as a Mining Engineer for a total of 19 years since my graduation from university. My relevant experience includes contributions to numerous feasibility, pre-feasibility, preliminary assessment and competent person reports while employed with SRK, Denver. Previously, I worked on the design and implementation of mine planning and scheduling systems, long term mine design with environmental focus, and mine planning corporate standards for Solid Energy, New Zealand. In addition, I have worked in various sales and support roles utilizing Vulcan Software and MineSuite Production Statistics where I gained considerable exposure to mining operations and projects around the world.
4. 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 a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Montagne d'Or property on April 1 to 3, 2014.
6. I am responsible for the preparation of mine design and mine planning Sections 15, 16, 18, 23, 24, 27 and 28 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the technical report titled, "NI 43-101 Preliminary Assessment, Golden Star Resources Ltd., Paul Isnard Project, French Guiana" with an Effective Date of February 29, 2008, prepared for Golden Star Resources Ltd.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 31st Day of July, 2015.

"Signed and Sealed"

Bret C. Swanson, BEng Mining, MAusIMM, MMSAQP

SRK Denver
7175 West Jefferson Avenue.
Suite 3000
Lakewood, CO 80235

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Eric J. Olin, MSc Metallurgy, MBA, SME-RM, MAusIMM do hereby certify that:

1. I am a Principal Consultant (Metallurgy) of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015, prepared for Nord Gold N.V. with Columbus Gold Corporation (the "Technical Report").
3. I graduated with a Master of Science degree in Metallurgical Engineering from the Colorado School of Mines in 1976. I am a Registered Member of The Society for Mining, Metallurgy and Exploration, Inc. I have worked as a Metallurgist for a total of 31 years since my graduation from the Colorado School of Mines. My relevant experience includes extensive consulting, plant operations, process development, project management and research & development experience with base metals, precious metals, ferrous metals and industrial minerals. I have served as the plant superintendent for several gold and base metal mining operations. Additionally, I have been involved with numerous third-party due diligence audits, and preparation of project conceptual, pre-feasibility and full-feasibility studies.
4. 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 a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I did not visit the Montagne d'Or property.
6. I am responsible for the preparation of mineral processing, metallurgy and recovery Sections 13 (except 13.10) and 17, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the technical report titled "NI 43-101 Technical Report on Updated Resources, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of April 11, 2015.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 31st Day of July, 2015.

"Signed and Sealed"

____

Eric J. Olin, MSc Metallurgy, MBA, SME-RM, MAusIMM

SRK Denver
7175 West Jefferson Avenue.
Suite 3000
Lakewood, CO 80235

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Grant A. Malensek, MEng, PEng/PGeo, do hereby certify that:

1. I am Principal Consultant (Mineral Project Evaluation) of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015, prepared for Nord Gold N.V. with Columbus Gold Corporation (the "Technical Report").
3. I graduated with a degree in B.S. Geological Sciences from University of British Columbia in 1987. In addition, I have obtained a M.E. in Geological Engineering (Colorado School of Mines, 1997). I am a Professional Engineer of the Association of Professional Engineers & Geoscientists of British Columbia. I have worked as an Engineer for a total of over 20 years since my graduation from university. My relevant experience includes business experience in financial analysis, project management and business development.
4. 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 a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I did not visit the Montagne d'Or property.
6. I am responsible for the preparation of Sections 13.10, 19, 21, and 22 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 31st Day of July, 2015.

"Signed and Sealed"

____
Grant A. Malensek, MEng, PEng/PGeo

SRK Consulting (U.S.) Inc.
5250 Neil Road, Suite 300
Reno, Nevada 89502

T: (775) 828-6800
F: (775) 828-6820

reno@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Mark Willow, MSc, CEM, SME-RM do hereby certify that:

1. I am Principal Environmental Scientist of SRK Consulting (U.S.), Inc., 5250 Neil Road, Reno, Nevada 89511.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015, prepared for Nord Gold N.V. with Columbus Gold Corporation (the "Technical Report").
3. I graduated with Bachelor's degree in Fisheries and Wildlife Management from the University of Missouri in 1987 and a Master's degree in Environmental Science and Engineering from the Colorado School of Mines in 1995. I have worked as Biologist/Environmental Scientist for a total of 22 years since my graduation from university. My relevant experience includes environmental due diligence/competent persons evaluations of developmental phase and operational phase mines through the world, including small gold mining projects in Panama, Senegal, Peru and Colombia; open pit and underground coal mines in Russia; several large copper mines and processing facilities in Mexico; and a mine/coking operation in China. My Project Manager experience includes several site characterization and mine closure projects. I work closely with the U.S. Forest Service and U.S. Bureau of Land Management on several permitting and mine closure projects to develop uniquely successful and cost effective closure alternatives for the abandoned mining operations. Finally, I draw upon this diverse background for knowledge and experience as a human health and ecological risk assessor with respect to potential environmental impacts associated with operating and closing mining properties, and have experienced in the development of Preliminary Remediation Goals and hazard/risk calculations for site remedial action plans under CERCLA activities according to current U.S. EPA risk assessment guidance. I am a Certified Environmental Manager (CEM) in the State of Nevada (#1832) in accordance with Nevada Administrative Code NAC 459.970 through 459.9729. Before any person consults for a fee in matters concerning: the management of hazardous waste; the investigation of a release or potential release of a hazardous substance; the sampling of any media to determine the release of a hazardous substance; the response to a release or cleanup of a hazardous substance; or the remediation soil or water contaminated with a hazardous substance, they must be certified by the Nevada Division of Environmental Protection, Bureau of Corrective Action;
4. 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 a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Montagne d'Or property on April 1 to 3, 2014.
6. I am responsible for the preparation of environmental studies, permitting and social or community impact Section 20 and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 31st Day of July, 2015.

"Signed and Sealed"

Mark Willow, MSc, CEM, SME-RM