B2GOLD CORP

NI 43-101 TECHNICAL REPORT

FEASIBILITY STUDY ON THE FEKOLA GOLD PROJECT IN MALI

Qualified Persons:

Mr T Garagan, P.Geo, B2Gold

Mr P Montano, P.E., B2Gold

Mr W Lytle, P.E., B2Gold

Mr K Jones, P.E., B2Gold

Mrs S Hunter, Principal Process Engineer, Lycopodium Minerals

Mr D Morgan, Managing Director, Knight Piésold

3174-STY-002

June 30, 2015

File Location: 16.04

Rev: 0

NI 43-101 TECHNICAL REPORT
FEASIBILITY STUDY ON THE FEKOLA GOLD PROJECT IN MALI

3174-STY-002

Table of Contents

			Page
1.0	**SUMMARY**		**1.3**
	1.1	Introduction	1.3
	1.2	Property Description and Location	1.4
		1.2.1 Project Ownership and Mineral Tenure	1.5
	1.3	Accessibility, Climate, Local Resources, Infrastructure and Physiography	1.5
	1.4	History	1.6
	1.5	Geology and Mineralisation	1.7
	1.6	Exploration	1.8
	1.7	Drilling	1.8
	1.8	Sample Preparation, Analyses and Security	1.10
	1.9	Data Verification	1.10
	1.10	Mineral Processing and Metallurgical Testing	1.10
	1.11	Mineral Resource	1.12
	1.12	Mineral Reserve	1.13
	1.13	Mining	1.13
		1.13.2 Mine Design	1.15
		1.13.3 Mine Production	1.15
		1.13.4 Mine Operations	1.15
	1.14	Recovery Methods	1.16
	1.15	Infrastructure	1.20
		1.15.2 Roads	1.22
		1.15.3 Airstrip	1.22
		1.15.4 Tailings Disposal (Tailings Storage Facility)	1.22
		1.15.5 Surface Water Management	1.23
		1.15.6 Falémé River Flood Assessment	1.23
		1.15.7 Water Supply	1.23
		1.15.8 Power Supply and Distribution	1.24
		1.15.9 Fuel Supply	1.25
		1.15.10 Sewage and Solid Waste Management	1.25
		1.15.11 Accommodation Camp	1.25
		1.15.12 Buildings and Support Facilities	1.25
	1.16	Market Studies and Contracts	1.26
	1.17	Required Permits and Environmental Considerations	1.27
		1.17.1 Environmental Consideration	1.27
		1.17.2 Permit Consideration	1.27
	1.18	Capital and Operating Costs	1.27
		1.18.1 Operating Costs	1.27
		1.18.2 Capital Costs	1.28
		1.18.3 Deferred Capital	1.29
		1.18.4 Project Implementation	1.29
	1.19	Economic Analysis	1.30
	1.20	Adjacent Properties	1.30
	1.21	Conclusions	1.31
	1.22	Recommendations	1.31
2.0	**INTRODUCTION**		**2.1**
	2.1	Introduction	2.1
	2.2	Terms of Reference	2.2
	2.3	Qualified Persons	2.3

Table of Contents

			Page
2.4	Site Visits and Scope of Personal Inspection		2.3
2.5	Effective Dates		2.4
	2.5.1	Information Sources and References	2.4
	2.5.2	Previous Technical Reports	2.4
3.0	**RELIANCE ON OTHER EXPERTS**		**3.1**
3.1	General Statement		3.1
4.0	**PROPERTY DESCRIPTION AND LOCATION**		**4.1**
4.1	Overview of the Property and Title in Mali		4.1
	4.1.1	Country Overview	4.1
	4.1.2	Mineral Tenure Regime	4.1
	4.1.3	State Participation	4.3
	4.1.4	Surface Rights	4.4
	4.1.5	Royalty and Mining Tax Considerations	4.4
	4.1.6	Environmental Considerations	4.4
	4.1.7	Social Considerations	4.5
	4.1.8	Taxation Considerations	4.5
	4.1.9	Policy Perception Index	4.6
4.2	Project Ownership		4.6
	4.2.1	Médinandi Exploitation License Ownership	4.6
	4.2.2	Songhoi Legal Standing	4.8
	4.2.3	Establishment Convention	4.8
	4.2.4	Mali Government Interest	4.9
	4.2.5	Option Agreements	4.10
	4.2.6	Legal Proceedings	4.10
4.3	Mineral Tenure		4.10
	4.3.1	Médinandi Exploitation License	4.10
	4.3.2	Exploration Permits	4.13
4.4	Additional Tenure in Mali		4.14
4.5	Surface Rights		4.14
4.6	Water Rights		4.14
4.7	Royalties and Encumbrances		4.15
4.8	Property Agreements		4.15
4.9	Permits		4.15
4.10	Environmental Liabilities		4.15
4.11	Social License		4.15
4.12	Comments on Section 4		4.15
5.0	**ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY**		**5.1**
5.1	Accessibility		5.1
5.2	Climate		5.1
5.3	Local Resources and Infrastructure		5.1
5.4	Physiography, Topography, Elevation and Vegetation		5.2
5.5	Comments on Section 5		5.3
6.0	**HISTORY**		**6.1**
6.1	Ownership History		6.1
	6.1.1	Legacy Period (1954 to 2001)	6.1

Table of Contents

			Page
		6.1.2 Recent Period (2006 to Present)	6.1
	6.2	Exploration History	6.2
		6.2.1 Legacy Period	6.2
		6.2.2 Recent Period	6.3
	6.3	Resource History	6.6
7.0	**GEOLOGICAL SETTING, MINERALIZATION AND ALTERATION**		**7.1**
	7.1	Regional Geology	7.1
		7.1.1 Overview	7.1
		7.1.2 Regional Stratigraphy	7.1
		7.1.3 Regional Structures	7.3
	7.2	Deposit Geology	7.3
		7.2.2 Deposit Lithology	7.5
	7.3	Deposit Scale Geology	7.8
		7.3.1 D_1 and D_2 Structures	7.8
		7.3.2 Brittle-ductile Structures	7.11
		7.3.3 Late Brittle Structures – Fekola Fault	7.13
	7.4	Mineralization and Alteration	7.14
8.0	**DEPOSIT TYPES**		**8.1**
	8.1	Deposit Model	8.1
	8.2	Comparison of Fekola to Loulo Gold Deposits	8.2
9.0	**EXPLORATION**		**9.1**
	9.1	Grids and Surveys	9.1
	9.2	Geological Mapping	9.1
	9.3	Geochemical Sampling	9.2
	9.4	Geophysics	9.5
	9.5	Pits and Trenches	9.8
	9.6	Petrology, Mineralogy and Research Studies	9.8
		9.6.1 Schandl (2012)	9.8
		9.6.2 Pathfinder (2012 to 2013)	9.9
		9.6.3 Panterra Geoservices (2014)	9.9
	9.7	Exploration Potential	9.9
		9.7.1 Fekola Deposit	9.9
		9.7.2 Exploration Targets	9.9
	9.8	Comments on Section 9	9.12
10.0	**DRILLING**		**10.1**
	10.1	Introduction	10.1
	10.2	Legacy Drilling	10.1
	10.3	Drill Data for January 2013 PEA Mineral Resource Estimate (as at 15 January 2013)	10.8
	10.4	Drill Data for August 2013 Mineral Resource Estimate Update (as at 5 August 2013)	10.10
	10.5	Drill Data for February 2015 Mineral Resource Estimate Update (as at January 24, 2015)	10.10
	10.6	Drill contractors	10.11
	10.7	Papillon / B2Gold Drill Methods	10.11
		10.7.1 Auger, Rotary Air Blast and Aircore	10.11

Table of Contents

Page

10.7.2 Reverse Circulation 10.11
10.7.3 Core Drilling 10.12
10.8 Papillon/B2Gold Geological Logging 10.12
10.8.2 RC 10.14
10.8.3 Core 10.15
10.8.4 Exploration-stage Geotechnical Logging 10.17
10.8.5 Magnetic Susceptibility 10.17
10.9 Recovery 10.17
10.10 Collar Surveys 10.17
10.11 Downhole Surveys 10.18
10.11.1 RC 10.18
10.11.2 Core 10.18
10.12 Condemnation, Geotechnical and Hydrogeological Drilling 10.18
10.13 Metallurgical Drilling 10.21
10.14 Sample Length / True Thickness 10.21
10.15 Drilling Since Completion of Resource Estimates 10.25
10.16 Comments on Section 10 10.27

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY 11.1
11.1 Sample Preparation and Analysis 11.1
11.1.1 RC Drilling Samples 11.1
11.1.2 Diamond Drilling Samples 11.1
11.1.3 Metallurgical Sampling 11.2
11.1.4 Analytical and Test Laboratories 11.2
11.2 Quality Assurance and Quality Control Programmes 11.5
11.2.2 Standards 11.5
11.2.3 Blanks 11.5
11.2.4 Duplicate Samples 11.6
11.2.5 Magnetic Susceptibility 11.7
11.2.6 Density 11.7
11.3 ICP Analysis 11.7
11.4 Density Analysis 11.7
11.4.1 Density by Depth 11.8
11.4.2 Density by Rock Type 11.9
11.4.3 Density by Mineralized Zones 11.9
11.5 Portable X-ray Fluorescence Sampling 11.9
11.6 Sample Security 11.10
11.7 Sample Storage 11.10
11.8 Project Drillhole Database 11.10
11.8.1 Procedures Prior to January 2015 11.10
11.8.2 Procedures Post-January 2015 11.12
11.9 Comments on Section 11 11.13

12.0 DATA VERIFICATION 12.1
12.1 QAQC Data Reviews 12.1
12.1.1 Papillon / B2Gold Data Checks 12.1
12.1.2 ioGlobal Data Checks 12.4
12.1.3 Results 12.4
12.2 QAQC Analysis 12.4

Table of Contents

Page

12.2.1 MPR Reviews, Mineral Resource Estimate Update Data Support 12.4
12.2.2 B2Gold QAQC Review 12.6
12.2.3 Comments on Section 12 12.18

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING 13.1
13.1 Introduction 13.1
13.2 Metallurgical Samples 13.2
13.2.1 Geology 13.2
13.2.2 Metallurgical Domains 13.4
13.2.3 Metallurgical Samples 13.6
13.2.4 Comminution Samples 13.8
13.3 Metallurgical Testing 13.8
13.3.1 Metallurgical Testwork Programme 13.8
13.3.2 Head Analysis 13.13
13.3.3 Mineralogy 13.19
13.3.4 Comminution Testwork 13.20
13.3.5 Gravity Concentration 13.22
13.3.6 Grind / Recovery Testwork 13.23
13.3.7 Preg Robbing Assessment 13.24
13.3.8 Whole Ore Leach Cyanidation Optimisation Testwork 13.25
13.3.9 Whole Ore Cyanidation of Variability Samples at Optimised Leach Conditions 13.33
13.3.10 Diagnostic Leaching Testwork 13.36
13.3.11 Bulk Cyanidation Tests 13.37
13.3.12 Cyanide Destruction Testwork 13.37
13.3.13 Oxygen Uptake Testwork 13.38
13.3.14 Carbon Modelling 13.40
13.3.15 Slurry Rheology Testwork 13.43
13.3.16 Thickening and Flocculation Testwork 13.44
13.3.17 Materials Handling Testwork 13.45
13.4 Comminution Circuit Selection 13.46
13.4.1 Comminution Circuit Design Basis 13.46
13.4.2 Comminution Circuit Modelling 13.46
13.4.3 Comminution Circuit Selection 13.46
13.5 Metallurgical Testwork Summary 13.47
13.5.1 Selected Treatment Route 13.47
13.5.2 Key Process Design Parameters 13.47
13.5.3 Metallurgical Recoveries 13.48
13.5.4 Reagent Consumption 13.50
13.5.5 Testwork Conclusions 13.50

14.0 MINERAL RESOURCE ESTIMATE 14.1
14.1 Summary 14.1
14.2 Software Used 14.2
14.3 Available Data 14.2
14.4 Exploratory Data Analysis 14.3
14.5 Geological Interpretation and Modelling 14.4
14.5.1 Structural Model 14.5
14.5.2 Lithological and Pyrite Models 14.5

Table of Contents

			Page
	14.5.3	Weathering Domains	14.5
	14.5.4	Mineralization Domains	14.8
	14.5.5	Contact Analysis	14.15
	14.5.6	High Grade Shoot	14.16
14.6		Evaluation of Outliers, Compositing and Variography	14.16
	14.6.1	Evaluation of Outliers	14.16
	14.6.2	Composites	14.17
	14.6.3	Variography	14.17
14.7		Resource Block Model	14.21
	14.7.1	Block Model Dimensions	14.21
	14.7.2	Gold Grade Estimation	14.22
	14.7.3	Discussion of Re-blocking	14.22
	14.7.4	Bulk Density	14.23
14.8		Model Validation	14.23
	14.8.1	Visual Comparison of Composites to Block Grade Estimates	14.23
	14.8.2	Comparison Statistics	14.26
	14.8.3	Swath Plots	14.26
	14.8.4	Comparison to Previous Resource Models	14.29
14.9		Mineral Resource Classification	14.31
	14.9.1	Reasonable Prospects of Eventual Economic Extraction	14.31
14.10		Mineral Resource Statement	14.31
	14.10.2	Sensitivity of Mineral Resource to Cut-off Grade	14.32
14.11		Recommendations	14.33
14.12		Comments on Section 14	14.33
15.0		**MINERAL RESERVE ESTIMATE**	**15.1**
15.1		Summary of Mineral Reserve Estimate	15.1
15.2		Basis of Estimate	15.1
15.3		Pit Optimisation Key Assumptions	15.2
	15.3.1	Resource Model	15.2
	15.3.2	Dilution and Ore Recovery	15.3
	15.3.3	Geotechnical Considerations	15.3
	15.3.4	Base Mining Cost Estimate	15.4
	15.3.5	Throughput Costs	15.5
	15.3.6	Gold Price, Royalty and Discounting	15.6
	15.3.7	Process Recovery and Cutoff Grades	15.6
15.4		Pit Optimization Results	15.7
	15.4.1	Methodology	15.7
	15.4.2	Whittle Results and Pit Shell Selection	15.7
	15.4.3	Pit Optimization Sensitivity Results	15.9
15.5		Mine Design Criteria	15.12
	15.5.1	Pit Development Strategy	15.12
	15.5.2	Pit Design Criteria	15.12
	15.5.3	Road and Ramp Design Criteria	15.13
	15.5.4	Waste Storage Facility Design Criteria	15.13
15.6		Pit Stage Designs	15.13
15.7		Pit Stage Mining Inventory	15.18
	15.7.1	Cut-off Grades for Ore Classification	15.18
	15.7.2	Mining Inventory by Cut-off Grade and Pit Stages	15.18
15.8		Waste Storage Facility Designs and Site Layout	15.19

Table of Contents

			Page
	15.9	Risk Assessment	15.21
16.0	**MINING**		**16.1**
	16.1	Mining Method	16.1
		16.1.1 General Definition	16.1
		16.1.2 Shifts and Personnel	16.1
		16.1.3 Mining Equipment	16.2
		16.1.4 Grade Control	16.3
		16.1.5 Ore Handling	16.3
	16.2	Drilling Blasting	16.4
	16.3	Pit Dewatering	16.4
	16.4	Environmental Monitoring and Control	16.5
	16.5	Mine Production Schedule	16.5
	16.6	Mining Risk Assessment	16.9
	16.7	Limitations and Recommendations	16.9
17.0	**RECOVERY METHODS**		**17.1**
	17.1	Process Design	17.1
		17.1.1 Selected Process Flowsheet	17.1
		17.1.2 Key Process Design Criteria	17.4
	17.2	Process and Plant Description	17.5
		17.2.1 Ore Receiving and Crushing	17.5
		17.2.2 Crushed Ore Stockpile	17.6
		17.2.3 Grinding and Classification	17.6
		17.2.4 Pebble Crushing	17.8
		17.2.5 Leach Thickening	17.8
		17.2.6 Carbon in Columns Circuit	17.9
		17.2.7 Leach Circuit	17.10
		17.2.8 Carbon in Pulp Circuit	17.11
		17.2.9 Acid Wash, Elution, Electrowinning and Gold Room	17.12
		17.2.10 Carbon Regeneration	17.15
		17.2.11 Cyanide Destruction	17.15
		17.2.12 Tailings Thickening and Disposal	17.16
		17.2.13 Reagents Mixing and Storage	17.16
		17.2.14 Water Services	17.20
		17.2.15 Air Services	17.22
	17.3	Plant Consumption	17.22
		17.3.1 Energy Consumption	17.22
		17.3.2 Water Consumption	17.23
		17.3.3 Reagent and Consumable Consumption	17.23
	17.4	Plant Control System	17.24
18.0	**PROJECT INFRASTRUCTURE**		**18.1**
	18.1	Overall Site Development	18.1
	18.2	Roads	18.3
		18.2.1 Access to Site	18.3
		18.2.2 Project Site Roads	18.3
		18.2.3 Access Tracks	18.3
	18.3	Airstrip	18.3
	18.4	Tailings Storage Facility (TSF)	18.4

Table of Contents

			Page
	18.4.1	Capacity and Location	18.4
	18.4.2	Design Considerations	18.4
	18.4.3	Design Summary	18.4
	18.4.4	Monitoring	18.11
	18.4.5	Rehabilitation	18.11
	18.4.6	Tailings Geochemistry	18.11
	18.4.7	Geotechnical	18.13
	18.4.8	Seepage Assessment	18.13
	18.4.9	Stability Analysis	18.14
	18.4.10	Decant from Tailings Storage Facility	18.15
18.5		Surface Water Management	18.15
	18.5.1	Design Objectives	18.15
	18.5.2	Diversion Structures	18.16
	18.5.3	Collection and Control Structures	18.16
	18.5.4	Contact Water Storage Pond	18.19
	18.5.5	Sediment Control Structures	18.19
	18.5.6	Falémé River Flood Assessment	18.19
18.6		Water Supply	18.21
	18.6.1	Pit Groundwater	18.21
	18.6.2	Surface Water	18.22
	18.6.3	Falémé River Bores	18.22
	18.6.4	Water Balance	18.22
18.7		Power	18.23
	18.7.1	Power Generation	18.23
	18.7.2	Electrical Distribution	18.23
	18.7.3	Electrical Buildings	18.24
	18.7.4	Transformers and Compounds	18.24
18.8		Fuel Supply	18.25
18.9		Potable Water	18.25
	18.9.2	Process Plant	18.25
	18.9.3	Accommodation Camp	18.25
18.10		Sewage and Solid Waste Management	18.26
	18.10.1	Sewage Treatment	18.26
	18.10.2	Solid Wastes	18.26
18.11		Accommodation Camp	18.26
18.12		Plant Facilities	18.27
	18.12.1	General	18.27
	18.12.2	Mine Services Area Facilities	18.27
	18.12.3	Process Plant Area	18.28
	18.12.4	Other Support Facilities	18.28
19.0		**MARKET STUDIES AND CONTRACTS**	**19.1**
19.1		Market Studies	19.1
19.2		Contracts	19.1
20.0		**ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT**	**20.1**
20.1		Environmental Studies	20.1
20.2		Permitting	20.1
	20.2.1	Environmental Permit	20.1

Table of Contents

Page

20.2.2 Environmental Permit for Access Road 20.2
20.2.3 Mining Permit 20.2
20.2.4 Approval of Community Development Plan 20.2
20.2.5 Approval of Mine Closure Plan 20.2
20.2.6 Additional Permits and Authorizations 20.3
20.3 Environmental Impact Assessment Process 20.4
20.4 Summary of Environmental and Social Setting 20.5
20.4.1 Physical Setting 20.5
20.4.2 Biological Setting 20.9
20.4.3 Social Setting 20.10
20.5 Summary of Key Environmental and Social Potential Impacts 20.13
20.5.1 Air quality 20.14
20.5.2 Hydrology, Hydrogeology and Water Quality 20.15
20.5.3 Biodiversity 20.17
20.5.4 Access to Land Resources 20.19
20.5.5 Livelihood and Employment 20.21
20.5.6 Social Services / Infrastructure 20.22
20.6 Summary of Environmental and Social Management and Monitoring Plan 20.23
20.6.1 Environmental Monitoring Plan 20.24
20.6.2 Water Management Plan 20.24
20.6.3 Waste Management Plan 20.24
20.6.4 Erosion and Sediment Control Plan 20.25
20.6.5 Rehabilitation and Preliminary Mine Closure Plan 20.25
20.6.6 Emergency Preparedness and Response Plan 20.25
20.6.7 Biodiversity Action Plan 20.25
20.6.8 Stakeholder Engagement Plan 20.25
20.6.9 Community Development Plan 20.26
20.7 Mine Rehabilitation and Closure 20.26

21.0 CAPITAL AND OPERATING COSTS 21.1
21.1 Operating Costs 21.1
21.1.1 Mining Costs 21.1
21.1.2 Plant and Administration Costs 21.1
21.1.3 Power 21.2
21.1.4 Operating Consumables 21.2
21.1.5 Labour (Processing / Maintenance and Administration) 21.3
21.1.6 General and Administration Cost (excluding G&A labour) 21.3
21.1.7 Maintenance 21.4
21.1.8 Exclusions 21.4
21.2 Capital Cost Estimate 21.5
21.2.1 Summary 21.5
21.2.2 Estimating Currency and Base Date 21.6
21.2.3 Mining Capital Costs 21.7
21.2.4 Estimating Methodology 21.8
21.2.5 Quantity Development 21.8
21.2.6 Pricing Basis 21.9
21.2.7 Field Indirects 21.11
21.2.8 Engineering Services 21.12
21.2.9 Owner's Costs 21.12

Table of Contents

Page

21.2.10 Spares 21.12
21.2.11 Duties, taxes and Insurances 21.13
21.2.12 Contingency 21.13
21.2.13 Deferred Capital 21.14
21.2.14 Escalation 21.14
21.2.15 Qualifications and Assumptions 21.14
21.2.16 Exclusions 21.14
21.3 Deferred Capital 21.15
21.4 Project Implementation 21.16
21.4.1 Implementation Strategy 21.16

22.0 ECONOMIC ANALYSIS 22.1
22.1 Introduction 22.1
22.2 Summary 22.1
22.3 Principal Assumptions 22.2
22.3.1 Input Parameters and Project Schedule 22.2
22.3.2 Mineral Reserves and Mining 22.3
22.3.3 Production Schedule and Costs 22.3
22.3.4 US$ Capital Cost 22.6
22.4 US$ Financial Model Results 22.6
22.4.2 US$ 1,250 Au Case 22.10
22.5 Sensitivity Analysis 22.11

23.0 ADJACENT PROPERTIES 23.1
23.1 Regional Gold Deposits 23.1

24.0 OTHER RELEVANT DATA AND INFORMATION 24.1
24.1 Other Relevant Data 24.1
24.2 Risks and Opportunities 24.1
24.2.1 Identified Opportunities 24.1

25.0 CONCLUSIONS 25.1
25.1 Conclusions 25.1

26.0 RECOMMENDATIONS 26.1
26.1 Recommendations 26.1

27.0 SELECTED REFERENCES 27.1
27.1 References 27.1

28.0 QP CERTIFICATES 28.1

TABLES

Table 1.1.1 Study Contributors 1.3
Table 1.4.1 Summary Table of Historical Activities, Médinandi Exploitation Permit 1.6
Table 1.11.1 Fekola Mineral Resources, as at 24 January 2012 1.13
Table 1.12.1 Fekola Mineral Reserve Estimate as of 31 May 2015 1.13
Table 1.13.1 Mine Yearly Production Summary (Base Case) 1.15

Table of Contents

		Page
Table 1.13.2	Operations Equipment	1.16
Table 1.14.1	Summary of Key Process Design Criteria	1.19
Table 1.18.1	Fekola Mine Operating Cost Summary	1.28
Table 1.18.2	Fekola Process Plant 4 Mtpa Operating cost Summary	1.28
Table 1.18.3	Capital Estimate Summary (2Q15, ±15%)	1.29
Table 1.18.4	Deferred Capital Summary	1.29
Table 1.18.5	Key Project Milestones	1.30
Table 2.2.1	Exchange Rates	2.3
Table 4.2.1	2013 Fraser Institute Policy Perception Index Rankings, Africa	4.8
Table 4.3.1	Exploitation License Boundary Co-ordinates	4.12
Table 6.2.1	Summary Table of Historical Activities, Médinandi Exploitation Permit	6.4
Table 6.3.1	January 2013 Papillon Historical Resource Estimate	6.8
Table 6.3.2	August 2013 Papillon Historical Resource Estimate	6.9
Table 8.2.1	Comparison: Loulo Mining Camp Deposits to Fekola Deposit	8.4
Table 9.3.1	Summary Table, Geochemical Sampling, Médinandi Exploitation License	9.3
Table 9.4.1	Geophysical Surveys	9.5
Table 10.2.1	Drilling Summary, Médinandi Exploitation License, by Prospect Area	10.2
Table 10.2.2	Drilling Summary, Médinandi Exploitation License, by Year and Prospect Area (Current as at 24 January 2015)	10.3
Table 10.2.3	Drilling Summary, Médinandi Exploitation License, by Year (Current as at 24 January 2015)	10.5
Table 10.3.1	Summary of Fekola Drilling Database as at 15 January 2013 (Used for January 2013 PEA Mineral Resource)	10.8
Table 10.4.1	Summary of Fekola Drilling Database as at 5 August 2013 (Used for August 2013 Mineral Resource Estimate)	10.10
Table 10.5.1	Summary of Fekola Drilling Database as at 24 January 2015 (Used for February 2015 Mineral Resource Estimate)	10.11
Table 10.8.1	Geological Logging Codes	10.13
Table 10.14.1	Selected Drill Intercept Table	10.22
Table 11.1.1	Laboratory Summary	11.3
Table 11.1.2	Assay and Grade Population Distribution by Lab	11.3
Table 11.2.1	QAQC Insertion Frequency Summary	11.5
Table 11.4.1	Density by Depth	11.8
Table 11.4.2	Average Density by Lithology	11.9
Table 11.4.3	Average Density by Mineralized Zone	11.9
Table 12.2.1	Standard Analytical Bias	12.11
Table 12.2.2	Field Duplicate Grade Sums	12.12
Table 12.2.3	Field Duplicate Grade Sums by Grade Range Bin	12.13
Table 12.2.4	Umpire Population by Lab	12.15
Table 12.2.5	Umpire Population Grade Sums by Lab	12.16
Table 12.2.6	SGS_Kayes vs Bureau Veritas Umpire Grade Sums	12.18
Table 12.2.7	Bureau Veritas vs SGS_Kayes Bias	12.18
Table 13.2.1	Met Domains Gold Head Grade	13.4
Table 13.2.2	Fekola Deposit – Resource Tonnage Distribution across Met Domains	13.6
Table 13.2.3	Fekola Deposit – Resource Grade Distribution across Met Domains	13.6
Table 13.3.1	Summary of Metallurgical Testwork Programme	13.9
Table 13.3.2	Metallurgical Domain Composite Head Analysis	13.14
Table 13.3.3	Metallurgical Domain Composite Head Analysis (Part 1 of 3)	13.16
Table 13.3.4	Metallurgical Domain Composite Head Analysis (Part 2 of 3)	13.17

Table of Contents

		Page
Table 13.3.5	Metallurgical Domain Composite Head Analysis (Part 3 of 3)	13.18
Table 13.3.6	Bulk Mineralogy by QEMSCAN	13.19
Table 13.3.9	Gravity Separation Summary by SGS Lakefield	13.22
Table 13.3.10	Gravity Separation Summary by Dawson Met Lab	13.23
Table 13.3.11	Summary of Grind / Recovery Leach Results of Whole Ore Cyanidation	13.24
Table 13.3.12	Summary of Whole Ore vs CIL Cyanidation	13.25
Table 13.3.13	Summary of Whole Ore Cyanidation of Variability Samples at Optimised Leach Conditions	13.34
Table 13.3.14	Summary of Bulk Whole Ore Cyanidation Tests	13.37
Table 13.3.15	Carbon Adsorption of High Grade Domain Composite Sample	13.41
Table 13.3.16	Summary of Fekola Slurry Rheology	13.43
Table 13.3.17	Summary of Thickening Testwork Results	13.44
Table 13.5.1	Summary of Grinding, Leach and CIP Circuit Design Parameters	13.48
Table 13.5.2	Summary of Lime and Sodium Cyanide Addition	13.50
Table 14.1.1	Fekola Mineral Resources, as at 24 January 2015	14.2
Table 14.3.1	Drillhole Summary, Number and Meterage Used in Resource Model	14.3
Table 14.4.1	Statistics by Logged Percent Pyrite	14.3
Table 14.5.1	Weathering Domain Grouping	14.6
Table 14.5.2	Weathering Domain Codes	14.8
Table 14.5.3	Mineralization Domain Codes	14.8
Table 14.5.4	Gold Assay Statistics by Mineralization Domain Codes for Fresh Rock	14.13
Table 14.5.5	Gold Assay Statistics by Mineralization Domain Codes for Saprolite	14.14
Table 14.6.1	Mineralization Domain Codes	14.17
Table 14.6.2	Capped 2 m Gold Grade Composite Statistics by Mineralization Domain	14.17
Table 14.6.3	Variogram Models, All Mineralization Domains	14.18
Table 14.6.4	Variogram Models, Domain 101	14.18
Table 14.6.5	Variogram Models, Domain 105	14.19
Table 14.6.6	Variogram Models, Domain 110	14.19
Table 14.6.7	Variogram Models, Domain 125	14.20
Table 14.7.1	Block Model Specifications	14.21
Table 14.7.2	Block Model Fields	14.21
Table 14.7.3	Gold Grade Estimation Plan	14.22
Table 14.7.4	Densities used in Tonnage Calculations	14.23
Table 14.8.1	Comparison of Estimation Means at 0 g/t Cut-off	14.26
Table 14.8.2	Comparison of Modelling Methodologies, MPR August 2013 and B2Gold's February 2015 Model	14.30
Table 14.8.3	Comparison of B2Gold March 2015 Model to MPR August 2013 Model	14.30
Table 14.9.1	Parameters used for Resource Pit Optimization Runs	14.31
Table 14.10.1	Fekola Mineral Resources, as at 24 January 2015 [1,2,3,4,5,6]	14.32
Table 14.10.2	Sensitivity Analysis to Mineral Resources by Gold Cut-off Grades, as at 24 January 2015	14.32
Table 15.1.1	Mineral Reserve Estimate as of 31 May 2015	15.1
Table 15.3.1	Pit Optimisation Preliminary Operating Cost Estimates	15.5
Table 15.3.2	Process Recoveries and Cut-off Grade	15.7
Table 15.4.1	Base Case Pit Optimisation Results Summary	15.8
Table 15.4.2	Pit Optimisation Sensitivity Results Summary	15.10
Table 15.5.1	Pit Slope Design Parameters	15.12
Table 15.7.1	Mining Inventory Summary by Pit Stages	15.19

Table of Contents

		Page
Table 15.9.1	Project Development Risk Assessment	15.21
Table 16.1.1	Mine Personnel Requirements	16.2
Table 16.1.2	Mine Equipment Requirements	16.3
Table 16.2.1	Powder Factor Estimates (ANFO Equivalent)	16.4
Table 16.5.1	Production Schedule Overview	16.6
Table 16.5.2	Base Case Production Schedule Annual Summary	16.7
Table 16.6.1	Mining Operations Risk Assessment	16.9
Table 17.1.1	Summary of Key Process Design Criteria	17.4
Table 17.3.1	Average Power Demand Summary	17.23
Table 17.3.2	Annual Reagent and Consumable Consumption	17.24
Table 18.4.1	TSF Stage / Storage Capacity	18.9
Table 18.4.2	Desired Minimum Factors of Safety	18.14
Table 18.4.3	Summary of TSF Embankment Stability Analysis Results	18.14
Table 18.4.4	Average conditions – TSF recycle rates	18.15
Table 18.9.1	Potable Water Demand	18.25
Table 20.2.1	Summary of Additional Permits and Authorizations Required for the Project	20.3
Table 20.3.1	Specialist Studies Commissioned as part of the 2013 ESIS	20.4
Table 20.3.2	Specialist Studies Commissioned as part of the ESIA Update	20.5
Table 20.7.1	Summary of Estimated Rehabilitation and Closure Costs	20.27
Table 21.1.1	Fekola Mine Operating Cost Summary	21.1
Table 21.1.2	Fekola Mine Operating Cost per Mining Classification	21.1
Table 21.1.3	Fekola Process Plant 4 Mtpa Operating cost Summary	21.2
Table 21.1.4	Fekola Process Plant Power Cost by Plant Area	21.2
Table 21.1.5	Fekola Process Plant Consumables Cost by Plant Area	21.3
Table 21.1.6	Fekola Processing Plant and Admin Manning Levels	21.3
Table 21.1.7	Fekola Plant General and Administration Summary	21.4
Table 21.1.8	Fekola Plant Maintenance Materials Cost	21.4
Table 21.2.1	Capital Estimate Summary by Area (2Q15, ±15%)	21.6
Table 21.2.2	Capital Estimate Summary by Discipline (2Q15, ±15%)	21.6
Table 21.2.3	Foreign Currency Exposure	21.7
Table 21.2.4	Mining Capital Costs	21.8
Table 21.2.5	Derivation of Quantities	21.9
Table 21.2.6	Sources of Pricing	21.10
Table 21.2.7	Standard Direct Labour Gang Rates	21.11
Table 21.2.8	Contingency Percentage Summary	21.13
Table 21.3.1	Deferred Capital Summary	21.15
Table 21.3.2	Deferred Capital Schedule	21.15
Table 21.4.1	Key Project Milestones	21.16
Table 22.3.1	Financial Model Assumptions	22.2
Table 22.3.2	Doré Selling Costs	22.2
Table 22.3.3	Life of Mine Production Summary	22.5
Table 22.3.4	Sustaining Capital Summary (US$ Million)	22.6
Table 22.4.1	Overall Results of Financial Model US$	22.6
Table 22.4.2	Summary of Annual Cash Flow and Project Value	22.8
Table 22.4.3	Summary of Operating and Total Cash Costs US$	22.9
Table 22.4.4	Cash Flow and Project Value for US$ 1,250 Au Case	22.10
Table 22.5.1	Gold Price Sensitivity	22.12
Table 26.1.1	Key Project Milestones	26.1

NI 43-101 TECHNICAL REPORT
FEASIBILITY STUDY ON THE FEKOLA GOLD PROJECT IN MALI

3174-STY-002

Table of Contents

Page

FIGURES

Figure 1.2.1	Fekola Project Location	1.4
Figure 1.7.1	Fekola Deposit, Drillhole Location Plan as at February 2015	1.9
Figure 1.10.1	Gold Extraction Model	1.12
Figure 1.13.1	Fekola Pit Stage Design	1.14
Figure 1.14.1	Process Plant Layout	1.18
Figure 1.15.1	Overall Site Development Plan	1.21
Figure 2.1	Fekola Project Location	2.2
Figure 4.3.1	Mineral Tenure Boundary, Médinandi Exploitation License	4.11
Figure 4.3.2	Exploration Permit Tenure Plan	4.13
Figure 6.2.1	Prospect and Deposit Location Plan, Médinandi Exploitation Permit	6.6
Figure 7.1.1	Geology of the Kédougou–Kéniéba Inlier (modified from Lawrence, et al, 2013)	7.2
Figure 7.2.1	Plan map of Fekola Deposit Geology.	7.4
Figure 7.2.2	Principal Lithologies of the Fekola Deposit.	7.6
Figure 7.3.1	Laminated Siltstone / Mudstone Highly Transposed by S_1 Foliation into Strong S_0 / S_1 Composite Fabric	7.8
Figure 7.3.2	Isoclinal F_1 Intrafolial Folds in Layers of Siltstone and Mafic Volcaniclastic(?), FKD_154, 223.1 m	7.9
Figure 7.3.3	F_2 Folds in Banded Siltstone and Mudstone of the Turbidite Unit	7.9
Figure 7.3.4	Equal Area Plot of Poles to Bedding (BED), Contacts (CNT), Fold Surfaces (FOD) and Foliation (FOL)	7.10
Figure 7.3.5	North-looking Vertical Cross Section through the Fekola Deposit, Section 1387397 N	7.11
Figure 7.3.6	Hydrothermally Altered Cataclastic Breccia FKRD_11, 208m	7.12
Figure 7.3.7	Breccia Textured Dolomite-Albite-Quartz Veining, FKD_135, 270.8 and 271.3m	7.12
Figure 7.3.8	Late Brittle Structures Associated with the Fekola Fault Zone	7.13
Figure 7.4.1	Veinlet (A) and Disseminated-style (B) Pyrite Mineralization in Fine Grained Siltstone / Mudstone of the Turbidite Unit	7.15
Figure 7.4.2	East-west Profile of ICP-MS Analyses from ½ HQ Diameter Drill Core (n=1110)	7.15
Figure 8.1.1	Orogenic Deposits	8.1
Figure 9.2.1	Interpreted Geological Map from Geophysics and Drilling	9.2
Figure 9.3.1	Geochemical Anomaly Plan, Médinandi Exploitation License	9.4
Figure 9.4.1	Metal Factor from Gradient Array IP	9.6
Figure 9.4.2	Airborne EM Response (142 ms)	9.7
Figure 9.7.1	Exploration Potential (Drill Hole Grade Thickness)	9.11
Figure 10.2.1	Drill Collar Location Plan, Médinandi Exploitation License	10.6
Figure 10.2.2	Fekola Deposit, Drill Collar Location Plan	10.7
Figure 10.3.1	Fekola Deposit, Comparison Drill Collar Location Plan as at January, 2013 (Jan 2013 Model), August 2013 (Aug 2013 Model) and January 2015 (Feb 2015 Model)	10.9
Figure 10.8.1	Example Wet / Dry Chip Sample Recording	10.15
Figure 10.8.2	Example Wet / Dry Core Record	10.16
Figure 10.12.1	Location Plan, Geotechnical Drilling	10.20
Figure 10.15.1	Drilling Post 24 January 2015	10.26
Figure 11.1.1	Sample Preparation and Analysis Flowsheet (SGS)	11.4
Figure 11.2.1	Sample Preparation Flowsheet	11.6
Figure 11.4.1	Density Variation with Depth	11.8

Table of Contents

Page

Figure 11.8.1	Project Data Flowsheet	11.11
Figure 12.2.1	Fekola Pulp Duplicate Scatterplot	12.7
Figure 12.2.2	Fekola Pulp Dups Thompson-Howarth Plot	12.8
Figure 12.2.3	Fekola Lab Duplicate Scatterplot	12.9
Figure 12.2.4	Fekola Lab Dups Thompson-Howarth Plot	12.9
Figure 12.2.5	Fekola Blank Performance	12.10
Figure 12.2.6	Fekola Field Duplicate Scatterplot	12.13
Figure 12.2.7	Fekola Field Dups QQ Plot	12.14
Figure 12.2.8	Fekola Field Dups Thompson-Howarth Plot	12.15
Figure 12.2.9	Fekola Umpires: Kayes vs BV, BV vs Bamako	12.16
Figure 12.2.10	Fekola Umpires: BV vs Morila	12.17
Figure 12.2.11	Fekola Umpires: BV vs Kayes	12.17
Figure 13.2.1	Plan map of Fekola Deposit Geology	13.3
Figure 13.2.2	Plan View Location of Metallurgical Drillholes within Stage 7 Design Pit	13.5
Figure 13.2.3	Spatial Location of Metallurgical Drillholes within Stage 7 Design Pit	13.5
Figure 13.2.4	Longsection of Metallurgical Samples Origin	13.7
Figure 13.2.5	Longsection of Comminution Samples Origin	13.8
Figure 13.3.1	Metallurgical Testwork Programme Flow Chart	13.12
Figure 13.3.2	Effect of Residence Time	13.26
Figure 13.3.3	Effect of Lead Nitrate Addition	13.28
Figure 13.3.4	Effect of Cyanide Concentration	13.29
Figure 13.3.5	Effect of Dissolved Oxygen	13.30
Figure 13.3.6	Effect of Pulp Density	13.31
Figure 13.3.7	Reproducibility of Optimum Gold Leaching Conditions	13.32
Figure 13.3.8	Oxygen Demand of High Grade Domain Composite Sample	13.39
Figure 13.3.9	Oxygen Demand of Low Grade Domain Composite Sample	13.39
Figure 13.3.10	Oxygen Demand of Hanging Wall Domain Composite Sample	13.40
Figure 13.3.11	Carbon Adsorption of Bulk Leached HG Composite Sample	13.41
Figure 13.3.12	Equilibrium Loading of Gold onto Carbon from Bulk Leached HG Composite Sample	13.42
Figure 13.5.1	Gold Extraction Model	13.49
Figure 14.4.1	Gold Grade Distributions by Logged Lithology	14.4
Figure 14.5.1	Cross Section Looking North, Mineralization Domains	14.9
Figure 14.5.2	Longitudinal Section Looking West, Mineralization Domains	14.9
Figure 14.5.3	Level Plan, Mineralization Domains	14.10
Figure 14.5.4	Statistics of Logged Percent Pyrite by Mineralization Domain	14.11
Figure 14.5.5	Statistics of Gold Grade by Mineralization Domain	14.12
Figure 14.5.6	Contact Plot between Domain 101 and 105	14.15
Figure 14.5.7	Contact Plot between Domain 105 and 110	14.15
Figure 14.5.8	Contact Plot between Domain 110 and 125	14.16
Figure 14.8.1	Cross Section Looking North, Block Model and Composite Gold Grades	14.24
Figure 14.8.2	Plan View, Elevation 0 m, Block Model Gold Grades	14.25
Figure 14.8.3	Comparison of Grade Estimates by Northing	14.27
Figure 14.8.4	Comparison of Grade Estimates by Elevation	14.28
Figure 14.8.5	Comparison of Grade Estimates by Easting	14.29
Figure 15.3.1	Pit Slope Design Sectors	15.4
Figure 15.3.2	Gold Extraction v Head Grade (B2Gold, May 15)	15.6
Figure 15.4.1	Unit Production Cost per Ounce v Optimal Pit Resource Tonnage	15.9
Figure 15.4.2	Extent of Optimal Pit Shells by Gold Price	15.11

Table of Contents

		Page
Figure 15.6.1	Ultimate Pit Design	15.14
Figure 15.6.2	Plan View of Pit Stage Designs	15.15
Figure 15.6.3	Perspective and Section View of Pit Stage Designs	15.16
Figure 15.6.4	Northern Expansion of the Pit Limits in Six Stages	15.17
Figure 15.8.1	Waste Storage Facility Designs and Site Layout	15.20
Figure 16.5.1	Production Schedule Summary Graphs	16.8
Figure 17.1.1	Overall Process Flow Diagram	17.3
Figure 18.1.1	Overall Site Development Plan	18.2
Figure 18.4.1	TSF General Arrangement Stage 1	18.5
Figure 18.4.2	TSF General Arrangement Final Stage	18.6
Figure 18.4.3	TSF Typical Embankment Sections	18.7
Figure 18.4.4	TSF Typical Embankment Sections	18.8
Figure 18.5.1	Fresh and Contact Water Storage Ponds - General Arrangement	18.17
Figure 18.5.2	Fresh and Contact Water Storage Ponds - Typical Embankment Sections	18.18
Figure 20.4.1	Local Drainage Map of the Fekola Gold Project Area and Surrounds	20.8
Figure 20.4.2	West Sudanian Savannah Ecoregion (Yellow)	20.9
Figure 20.4.3	Proximity Map of Villages in the Area Surrounding the Project	20.12
Figure 22.5.1	Fekola Project Cash Flow Sensitivity	22.11
Figure 23.1.1	Regional Map of Known Gold Deposits	23.1

FEKOLA GOLD PROJECT

NI 43-101 - SUMMARY

3174-STY-002

Table of Contents

			Page
1.0	**SUMMARY**		**1.3**
	1.1	Introduction	1.3
	1.2	Property Description and Location	1.4
		1.2.1 Project Ownership and Mineral Tenure	1.5
	1.3	Accessibility, Climate, Local Resources, Infrastructure and Physiography	1.5
	1.4	History	1.6
	1.5	Geology and Mineralisation	1.7
	1.6	Exploration	1.8
	1.7	Drilling	1.8
	1.8	Sample Preparation, Analyses and Security	1.10
	1.9	Data Verification	1.10
	1.10	Mineral Processing and Metallurgical Testing	1.10
	1.11	Mineral Resource	1.12
	1.12	Mineral Reserve	1.13
	1.13	Mining	1.13
		1.13.2 Mine Design	1.15
		1.13.3 Mine Production	1.15
		1.13.4 Mine Operations	1.15
	1.14	Recovery Methods	1.16
	1.15	Infrastructure	1.20
		1.15.2 Roads	1.22
		1.15.3 Airstrip	1.22
		1.15.4 Tailings Disposal (Tailings Storage Facility)	1.22
		1.15.5 Surface Water Management	1.23
		1.15.6 Falémé River Flood Assessment	1.23
		1.15.7 Water Supply	1.23
		1.15.8 Power Supply and Distribution	1.24
		1.15.9 Fuel Supply	1.25
		1.15.10 Sewage and Solid Waste Management	1.25
		1.15.11 Accommodation Camp	1.25
		1.15.12 Buildings and Support Facilities	1.25
	1.16	Market Studies and Contracts	1.26
	1.17	Required Permits and Environmental Considerations	1.27
		1.17.1 Environmental Consideration	1.27
		1.17.2 Permit Consideration	1.27
	1.18	Capital and Operating Costs	1.27
		1.18.1 Operating Costs	1.27
		1.18.2 Capital Costs	1.28
		1.18.3 Deferred Capital	1.29
		1.18.4 Project Implementation	1.29
	1.19	Economic Analysis	1.30
	1.20	Adjacent Properties	1.30
	1.21	Conclusions	1.31
	1.22	Recommendations	1.31

TABLES

Table 1.1.1	Study Contributors	1.3
Table 1.4.1	Summary Table of Historical Activities, Médinandi Exploitation Permit	1.6

Table of Contents (Continued)

Page

Table 1.11.1	Fekola Mineral Resources, as at 24 January 2012	1.13
Table 1.12.1	Fekola Mineral Reserve Estimate as of 31 May 2015	1.13
Table 1.13.1	Mine Yearly Production Summary (Base Case)	1.15
Table 1.13.2	Operations Equipment	1.16
Table 1.14.1	Summary of Key Process Design Criteria	1.19
Table 1.18.1	Fekola Mine Operating Cost Summary	1.28
Table 1.18.2	Fekola Process Plant 4 Mtpa Operating cost Summary	1.28
Table 1.18.3	Capital Estimate Summary (2Q15, ±15%)	1.29
Table 1.18.4	Deferred Capital Summary	1.29
Table 1.18.5	Key Project Milestones	1.30

FIGURES

Figure 1.2.1	Fekola Project Location	1.4
Figure 1.7.1	Fekola Deposit, Drillhole Location Plan as at February 2015	1.9
Figure 1.10.1	Gold Extraction Model	1.12
Figure 1.13.1	Fekola Pit Stage Design	1.14
Figure 1.14.1	Process Plant Layout	1.18
Figure 1.15.1	Overall Site Development Plan	1.21

1.0 SUMMARY

1.1 Introduction

In January 2015, B2Gold Corp (B2Gold) engaged Lycopodium Minerals Pty Ltd (Lycopodium) to coordinate an Optimised Feasibility Study (OFS) for their Fekola Gold Project (the Project), located in south western Mali.

The OFS draws on previously published reports and includes new contributions from resources within B2Gold and other consulting groups as outlined in Table 1.1.1.

Table 1.1.1 Study Contributors

Section	Contributors and Referenced Reports	Qualified Person
Section 1 Executive Summary	Lycopodium	Sandy Hunter
Section 2 Introduction	Lycopodium	Sandy Hunter
Section 3 Reliance on Other Experts	Lycopodium	Sandy Hunter
Section 4 Location, Ownership & Permitting	B2Gold	William Lytle
Section 5 Accessibility, Climate, Local Resources, Infrastructure and Physiography	B2Gold	William Lytle
Section 6 History	B2Gold	Tom Garagan
Section 7 Geological Setting, Mineralisation and Alteration	B2Gold Geology	Tom Garagan
Section 8 Deposit Types	B2Gold Geology	Tom Garagan
Section 9 Exploration	B2Gold Geology	Tom Garagan
Section 10 Drilling	B2Gold Geology	Tom Garagan
Section 11 Sample Preparation, Analysis and Security	B2Gold Geology	Tom Garagan
Section 12 Data Verification	B2Gold Geology	Tom Garagan
Section 13 Mineral Processing and Metallurgical Testing	Lycopodium	Sandy Hunter
Section 14 Mineral Resource Estimate	B2Gold Geology	Tom Garagan
Section 15 Mineral Reserve Estimate	Mining Solutions Consultants	Peter Montano
Section 16 Mining Methods	Mining Solutions Consultants	Peter Montano
Section 17 Recovery Methods	Lycopodium	Sandy Hunter
Section 18 Infrastructure	Lycopodium / Knight Piésold/B2Gold	Sandy Hunter, David Morgan, Peter Montano
Section 19 Market Studies and Contracts	Lycopodium / B2Gold	Sandy Hunter, William Lytle
Section 20 Required Permits and Environmental Considerations	B2Gold	Ken Jones
Section 21 Capital and Operating Costs	Lycopodium / B2Gold	Sandy Hunter, Peter Montano
Section 22 Economic Analysis	B2Gold	William Lytle
Section 23 Adjacent Properties	B2Gold	Tom Garagan

Section	Contributors and Referenced Reports	Qualified Person
Section 24 Other Relevant Data and Information	Lycopodium	Sandy Hunter
Section 25 Conclusion	Lycopodium	Sandy Hunter
Section 26 Recommendations	Lycopodium	Sandy Hunter
Section 27 References	Lycopodium	Sandy Hunter

1.2 Property Description and Location

The Project is located within the Kayes Region, in south western Mali, on the border of Mali and Senegal. It is situated around 400 km west of the capital Bamako and 40 km south of the nearest city of Kéniéba.

The project area is within the Médinandi Exploitation License and located at (UTM-WGS84) 1,387,300N, 242,200E.

Figure 1.2.1 Fekola Project Location

1.2.1 Project Ownership and Mineral Tenure

A 75 km² mining lease (the Médinandi Exploitation License) was granted on 13 February 2014 under permit number 0070/PM-RM. The exploitation licence is current for a 30 year term, expiring February 2044.

The Médinandi Exploitation License is held in the name of Songhoi Resources Sàrl (Songhoi), of which B2Gold currently holds 100% of the shares. Upon the grant of an exploitation licence, the holder of the exploitation licence must take steps to create an exploitation company incorporated under the laws of Mail (the new Malian mining company). On creation of the new Malian mining company, B2Gold's interest in the project will be 90% with a contribution of 10% share of the ownership granted to the Government of Mali. Additionally up to a further 10% may be purchased by the Government of Mail for cash value. B2Gold is the Project operator.

Legal opinion provided supports that the Médinandi Exploitation License is valid and mineral tenure is sufficient to support declaration of Mineral Resources and Reserves.

Two exploration permits, Menankoto Sud and Bantako Est, held by Songhoi are within close proximity to the Médinandi tenement. The Menankoto Sud exploration permit is 52 km^2 in area and is located approximately 13 km to the north of the Médinandi Exploitation License. The permit is valid until 4 February 2017. The Bantako Est exploration permit is 57 km^2 in area and is located less than 5 km from the northeast corner of the Médinandi Exploitation License. The permit expired on 30 December 2014. The renewal was submitted as required and is currently awaiting Mali government signature.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to the Project is by road from either Dakar in Senegal or from the Mali capital of Bamako. The road from Bamako is approximately 480 km, along the recently completed Millennium Highway to Kéniéba, then a further 45 km on the new mine access road from Kéniéba to the project site. The new mine access road is currently under construction and will allow all-weather two-way traffic for construction and future mine operations.

Current air access is via charter flights to an airstrip 1 km east of Kéniéba, then by road to site. A new airstrip is currently under construction at the project site. The new airstrip will be an all-weather gravel runway designed to suit a Beech Craft 1900 or similar type of aircraft.

The Project is in the pre-Guinean climate zone, with an average annual rainfall of greater than 1,000 mm over a wet season lasting six months (May to October). Temperature variance is 10 – 46°C upon season, with a mean annual temperature of 28°C.

The project is in a remote location with limited existing infrastructure. Proposed project infrastructure includes water, power, communications, and waste and tailings disposal to allow a self-sufficient plant operating all year round.

The site is relatively flat with a mean elevation above sea level that varies from about 130 to 190 m. The Falémé River occupies a meandering drainage to the south and west of the project area.

1.4 History

Gold was first discovered in the area by a French prospector at the Fadougou Main Zone in 1953. Since this discovery and up until B2Gold acquired ownership of the project, various companies have held the exploration rights and conducted reconnaissance activities within the permit area. A summary of historical activities is provided in Table 1.4.1.

Table 1.4.1 Summary Table of Historical Activities, Médinandi Exploitation Permit

Company	Year	Work Completed
Prospector	1954 – 1955	Documentation of activities not available to B2Gold.
Sonarem	1962 – 1970	Appears to have included geological mapping, reconnaissance trenching and surface sampling, auger and core drilling, geophysical surveys. BRGM reportedly undertook a resource estimate. Work is not broken out in the information available as to which companies performed which activity. The programs identified a 10 km^2 zone which contained five gold anomalies related to a well defined structural feature. The two largest zones, noted as II and III, were considered to be potentially economic. The Médinandi gold deposit was interpreted to host three mineralized zones, oriented north–south and northeast–southwest, over an area of 1,200 m x 3.3 m.
BRGM	1975 – 1982	
Guefest	1992 – 1996	
WAG	1997 – 1998	Regional mapping, ground IP surveys, trenching, soil and termite geochemical sampling, auger and reverse circulation (RC) drilling, and resource estimates for the Fadougou Main Zone.
Randgold	1998 – 2001	Interpretation of Landsat and aeromagnetic data, geological and regolith mapping, regional geochemical soil and rock, compilation of data from previous work, and updated mineral resource estimate for the Fadougou deposit.
Central African / Songhoi	2006 – 2009	Mapping, soil geochemical surveys, IP and airborne magnetic and EM surveys over the project area, together with RC and core drilling (130 holes) over the Médinandi and Fadougou zones; updated resource estimate at Fadougou.
Colonial / Papillon / Songhoi	2010	110 RC holes (13,081 m) at Fadougou, Fekola and Tintiba; 162.9 line km of geophysical surveys; 300 termite samples from mounds located between Médinandi and Betakili; 114 RC holes (14,211 m) on 14 prospects around Fadougou and Fekola; 122 RAB holes (3,065 m) at Fekola.
	2011	Resistivity survey for underground potable water; pole–dipole ground geophysics, 17 east–west-oriented lines, with stations on 160 x 1,600 m spacing at Fekola; 1,546 soil samples at Médinandi East, Fadougou East, Fekola East; 107 aircore holes (1,782 m) at Betakili; 986 RAB holes (20,100 m) at Fadougou East, Fadougou NE, Fadougou ESE, and Betakili; 181 RC holes (12,449 m) at Fadougou E, Fadougou SE, Fadougou Central, Fekola; 14 core holes (2747.5 m) at Fekola.
	2012	1,132 soil samples at Médinandi north; ground geophysical surveys northwest of Médinandi and at Fekola; test gravimetric survey; collection of metallurgical samples for testwork; road access, camp and electrical supply upgrades; installation of automatic weather station; condemnation and sterilization drilling (59 holes for 6,325 m at Fadougou and Fekola); water bore drilling; LiDAR survey; geotechnical and environmental studies; 81 RC holes (10,271 m) at Fadougou SE and Fekola; 179 core holes (45,452 m); completion of a scoping level study.

Company	Year	Work Completed
	2013	Ground gravimetric test survey at Fekola north; road access and camp upgrades; continuation of environmental and geotechnical studies (including 20 core holes, 15 RC holes and 48 pits completed for geotechnical purposes); 18 boreholes (1,262 m) of hydrological drilling; one batch of 669 core samples (approximately 964.8 kg) and second batch of 822 core samples (approximately 1219.3 kg) collected and prepared for additional metallurgical testwork; 37 core holes (11,768 m) at Fekola, project wide drilling, including geotechnical holes totalling 274 holes (31,414 m) at Fekola and Fadougou SE; receipt on site of the run of mine laboratory equipment and container; completion of a pre feasibility study; environmental permit granted. Resource estimate for Fekola deposit supporting the Papillon PFS released to ASX on 24 January 2014. Updated resource estimate for the Fekola deposit released to the ASX on 3 September 2013.
	Q1 2014	Water sampling; 84 exploration drillholes (7,902 m); four geotechnical boreholes and 62 test geotechnical pits completed in the proposed TSF and plant areas, additional geotechnical work in the area of the east wall of the proposed pit, packer testing; XRF analysis of selected core intervals from five core holes; magnetic susceptibility measurements on 18 RC holes (5,940 samples); commencement of studies in support of detailed design; grant of exploitation license. Resource estimate for Fekola Satellites (Fadougou) updated.

In 2012, Papillon completed a scoping level study on the Médinandi deposit. The study outcomes indicated positive project economics which led to a further pre-feasibility study being conducted in 2013. Results were initially summarized in a report prepared in June 2013 with an update to this report presented by Papillon in a news release to the ASX on 26 June 2013 (the Papillon PFS).

The Papillon PFS indicated positive project economics under the assumptions in the study. As a consequence, Papillon commenced more detailed engineering and technical studies, and commenced some preliminary site works in support of future project construction, including site clearing activities.

In October 2014 B2Gold Corp. acquired 100% of the ordinary shares of Papillion Resources Limited by way of a merger agreement.

1.5 Geology and Mineralisation

The Fekola deposit is located in the eastern portion of the Paleoproterozoic Kédougou–Kéniéba Inlier which covers eastern Senegal and western Mali. The deposit is considered to be an example of an orogenic style gold deposit. Orogenic gold deposits dominantly form in metamorphic rocks in the mid to shallow crust, at or above the brittle-ductile transition, in compressional settings that facilitate transfer of hot gold bearing fluids from deeper levels. During a geological event, the rapid rise of fluids takes these out of equilibrium with their surroundings, resulting in gold precipitation.

The Fekola deposit is hosted by a moderate to steeply west dipping, folded sequence of marine meta-sediments of the Kofi group, which locally include: argillite and minor pelitic sediments; fine grained turbidites, comprising laminated to thin-bedded siliciclastic siltstone and mudstone; and a

heterolithic, mass flow breccia, or conglomerate. Fine grained thin marble units are present as a volumetrically minor rock type inter-bedded with the politic units. Minor mafic volcanic, or volcaniclastic units occur locally in the upper and possibly, lower portions of the hanging wall stratigraphy. Weakly feldspar-phyric felsic dykes are locally observed. The deposit has been subjected to upper greenschist facies metamorphism.

The Mineralization is broadly continuous and has been traced over a strike extent of approximately 1.5 km, to depths of up to 400 m below surface, with widths to 300 m. The greatest continuity is observed within a high grade shoot (>2 g/t Au) which plunges approximately 13° to the north-north west. Mineralization is open at depth, down plunge.

The majority of gold mineralisation in the Fekola deposit occurs in unweathered, fresh rock and is preferentially associated with stringers of pyrite parallel to the foliation and in fine disseminated pyrite. More specifically, the gold mineralisation is associated with fine grained pyrite in tan dolomite-albite altered wall rocks, which locally contain diffuse, often deformed pale grey quartz-dolomite-pyrite-albite matrix veins and veinlets. Trace amounts of chalcopyrite have also been observed. The total sulphide content of the deposit is typically less than 5%.

1.6 Exploration

Exploration programs completed to date have been appropriate to the style of the Fekola deposit, leading to the discovery of a significant orebody. The Fekola deposit remains open at depth and the potential for extending the Fekola main deposit down plunge is good, particularly with the results obtained in the step out hole.

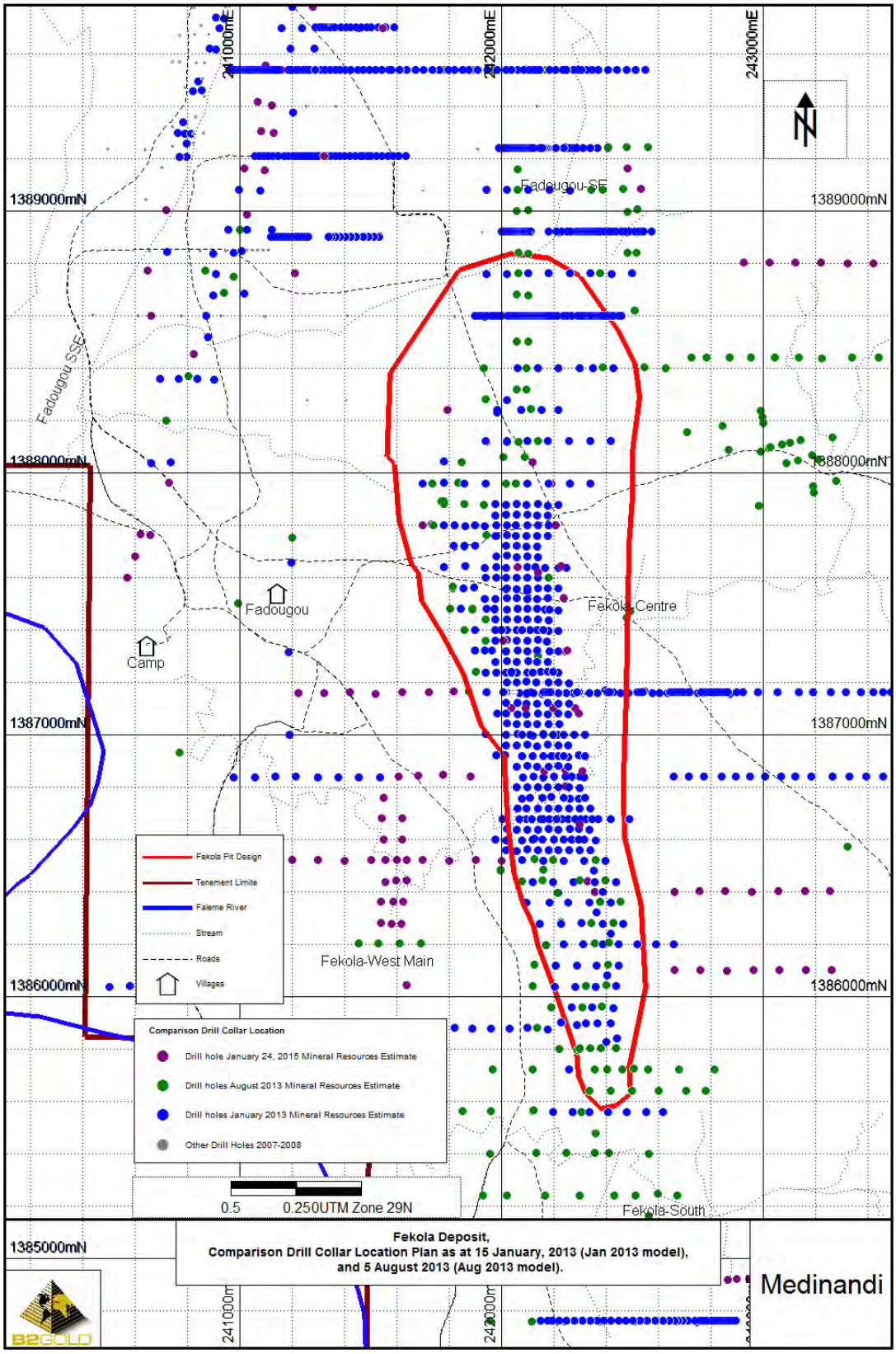
A better understanding of the Fekola deposit, its location in a regional and local context with respect to lithologies, alteration and structure, will aid in indicating areas of gold anomalism that has potential to support additional exploration focus.

1.7 Drilling

A total of 806 drillholes (143,372.98 m) were available at the data cutoff date of 24 January, 2015 for the February 2015 Mineral Resource estimate update.

The drillhole dataset within the Fekola Mineral Resource area is illustrated in plan view in Figure 1.7.1.

Figure 1.7.1 Fekola Deposit, Drillhole Location Plan as at February 2015



The quantity and quality of the lithological, collar and down hole survey data collected in the exploration and infill drill programs completed are sufficient to support Mineral Resource estimation and mine planning.

1.8 Sample Preparation, Analyses and Security

Sample collection, preparation, analysis and security for RC and core drill programs are in line with industry standard methods for gold deposits. Drill programs include insertion of blank, duplicate and standard reference material samples to monitor laboratory performance.

The primary laboratory used for sample analysis during January 2011 to June 2013 was SGS Kayes, Mali. From November 2013 to date the primary laboratory is SGS Bamako, Mali. The laboratories used are not formally accredited and currently there are no commercial mineral laboratories in the West African region that are fully accredited under ISO17025.

1.9 Data Verification

A reasonable level of verification has been completed during the work conducted to date, and no material issues have been left unidentified from the verification programs undertaken.

No problems with the database, sampling protocols, flowsheets, check analysis program, or data storage were identified that were sufficient to preclude the use of the database for estimation purposes. The quality of the gold analytical data is considered sufficiently reliable to support Mineral Resource estimation without limitations on Mineral Resource confidence categories. The data can be used in mine planning.

1.10 Mineral Processing and Metallurgical Testing

A metallurgical testwork programme has been completed to support the OFS of the Project. The majority of the programme was conducted by SGS Canada Inc. ('SGS Lakefield') in Lakefield, Ontario, Canada between 2014 and 2015. The remainder of the program was either completed at SGS Lakefield under supervision of a consultant or specialist or at another laboratory facility during the same time period.

The programme included testwork to establish:

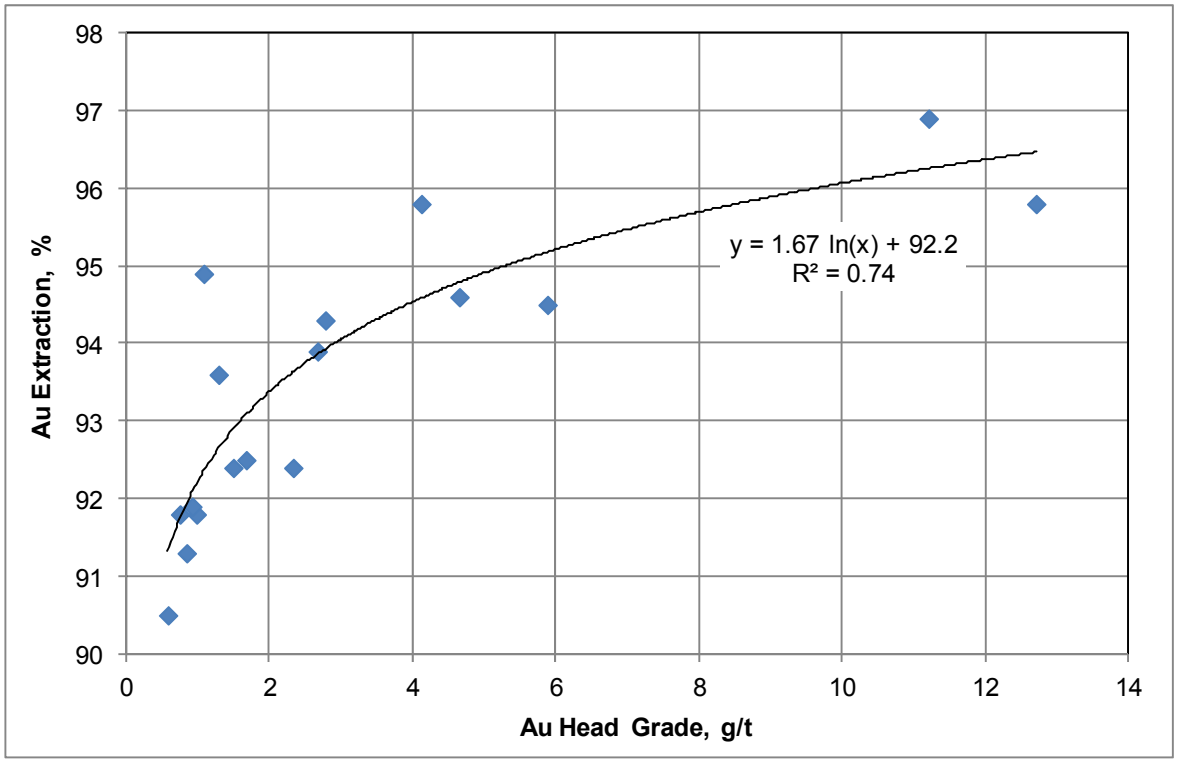
- Comminution characteristics of the ore.
- Materials handling properties.
- Mineralogy analysis.
- Sample head analysis.
- Gravity recoverable gold content.
- Optimum grind sizes for whole ore cyanidation leach.

- Optimum whole ore cyanidation leach conditions.
- Cyanide destruction testwork.
- Engineering data, including testing of oxygen uptake, slurry rheology, carbon kinetics and thickener settling rate.

Based on analysis of results the following conclusions can be drawn from the metallurgical and comminution testwork programmes:

- The Fekola deposit is classified as hard to very hard competency with above average grinding energy requirements and is moderate to highly abrasive. The ore is amenable to primary crushing followed by a SAG mill and ball mill with pebble crushing (SABC).
- Fekola ore is predominantly 'free milling', not 'preg robbing' and is amenable to gold extraction by conventional cyanidation.
- A gravity separation circuit is not warranted for the Fekola deposit. Instead, a carbon column adsorption circuit will be included to recover dissolved gold leached in the grinding circuit to facilitate early recovery of gold, particularly during high gold head grade periods.
- The optimum leaching conditions identified are 24 hour cyanidation with 350 ppm NaCN, initial lead nitrate addition of 100 g/t, pH 10.3 – 10.5, dissolved oxygen levels of ~15 ppm and a pulp density of 45% solids (w/w). The addition of lead nitrate and dissolved oxygen levels of 15 ppm is found to be beneficial in leach kinetics and overall recovery. Anticipated lime and cyanide addition rates are moderate.
- The ore typically yields good recoveries (87% to 97%). Testwork results show a logarithmic relationship between the measured gold head grade and resulting gold extraction under optimised leach conditions at a grind size of 75 microns. This relationship is graphically illustrated in Figure 1.10.1.
- Based on the absence of any preg robbing characteristics and very good adsorption properties, a CIP circuit has been selected for the Fekola process flowsheet.
- The cyanidation tailings responded well to cyanide destruction treatment using the SO_2 / Air process.
- The ore has a thickener specific settling rate of 0.3 m^2/tpd for both the leach and tailings thickener duties.

Figure 1.10.1 Gold Extraction Model



The overall relationship between gold head grade and gold extraction is shown by the following relationship:

Gold Extraction (%) = 1.6705 ln (Gold Head Grade (g Au/t)) + 92.218 [A]

1.11 Mineral Resource

The February 2015 Mineral Resource estimate forms the basis for the DFS and incorporates drilling results available at January 24, 2015.

The estimate was prepared in accordance with the definitions set out by the CIM Definition Standards for Mineral Resources and Mineral Reserves.

Mineral Resources are reported above a selected cutoff grade of 0.6 g/t gold and within a conceptual Lerchs–Grossmann pit shell. Parameters used for the Mineral Resource pit shell were based on preliminary work completed for the OFS (in March 2015), using a gold price of US$ 1,500/oz. Table 1.11.1 details the Fekola Mineral Resources, as at 24 January 2015.

Table 1.11.1 Fekola Mineral Resources, as at 24 January 2012

Cutoff Grade Au (g/t)	Indicated			Inferred		
	Tonnes x 1000	Au g/t	Ounces x 1000	Tonnes x 1000	Au g/t	Ounces x 1000
0.6	61,574	2.16	4,281	9,055	1.68	490

(1) **Mineral Resources are estimated using best practices as defined by the CIM and reporting of Mineral Resources is compliant and in accordance with the disclosure requirements of NI 43-101.**
(2) **The Qualified Person for the Mineral Resources is Tom Garagan, P. Geo.**
(3) **Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.**
(4) **Mineral Resources are inclusive of Mineral Reserves.**
(5) **Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an indicated or measured Mineral Resource as a result of continued exploration.**
(6) **Mineral Resource numbers have been rounded to reflect the accuracy of the estimate and numbers may not add due to rounding.**
(7) **Mineral Resources are reported above a 0.6 g/t gold grade cutoff and within a conceptual pit shell using a gold price of US$ 1,500/oz and preliminary operating costs as of March 2015.**
(8) **Mineral Resources are reported on a 100% ownership basis.**

1.12 Mineral Reserve

The mineral reserve estimate for the Project reported within the ultimate pit design is presented in Table 1.12.1

Table 1.12.1 Fekola Mineral Reserve Estimate as of 31 May 2015

Classification	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (Moz)
Probable Reserve	49.17	2.35	3.72

Notes:
(1) **CIM definitions were used for Mineral Reserves.**
(2) **The Qualified Person for the Mineral Reserves is Peter Montano, P.E.**
(3) **All Mineral Reserves were classified as Probable, based on the Indicated Mineral Resources.**
(4) **The Mineral Reserve cutoff grade is 0.8 g/t Au based on a US$ 1,300/oz gold price.**
(5) **The average metallurgical recovery is 92.8%, and the waste to ore ratio is 4.5:1 for Mineral Reserves.**
(6) **The block model was internally diluted at the SMU size of 5 m E x 20 m N x 10 m RL. In addition, a waste dilution factor of 5% was applied in conversion to the Mineral Reserves to account for mining factors.**
(7) **Sum of individual amounts may not equal due to rounding.**

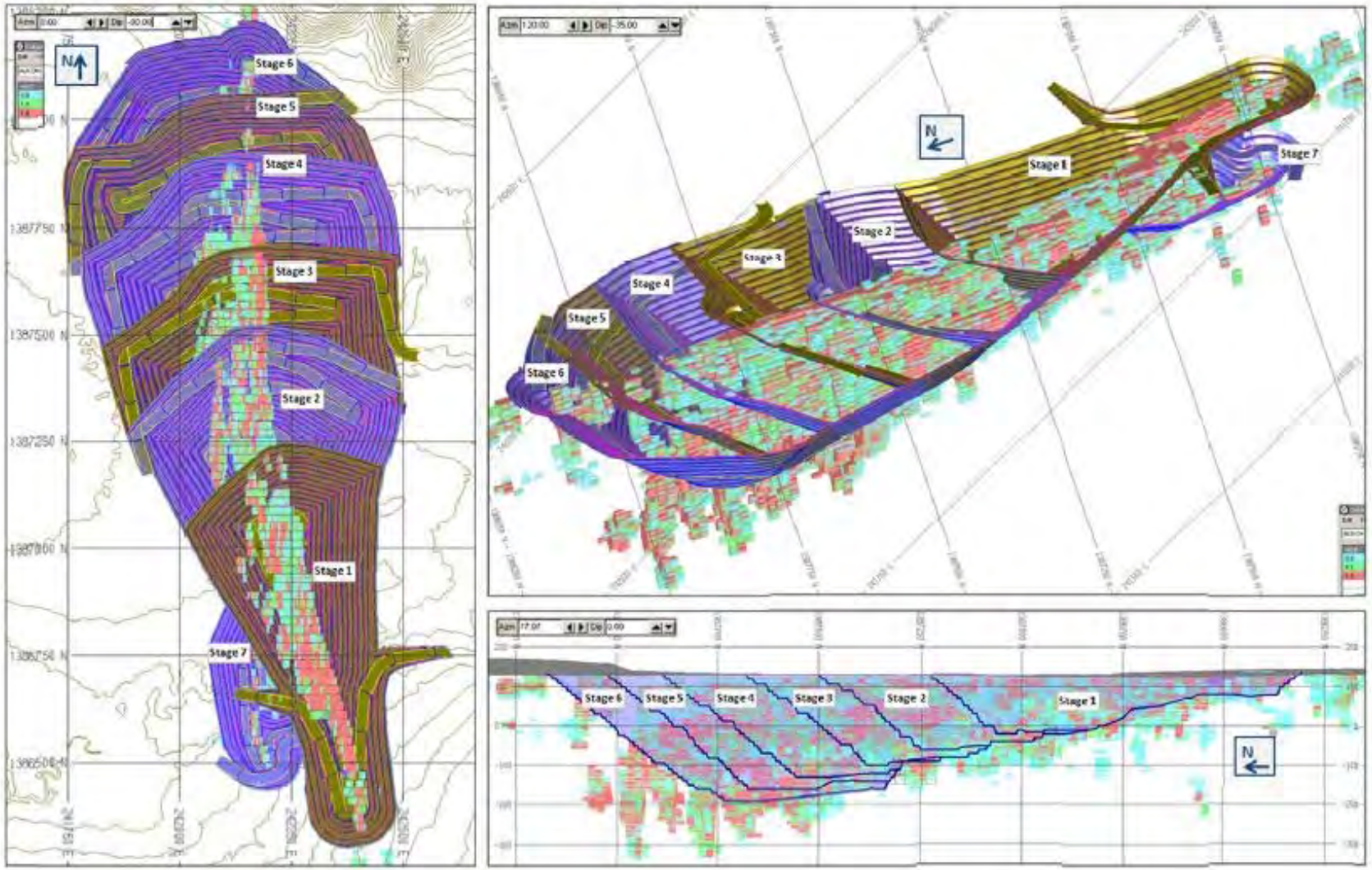
1.13 Mining

The mining study was completed by Mining Solutions Consultancy (MSC) in close liaison with B2Gold personnel.

The selected mining method for the Project is conventional open pit mining with the operations strategy based on owner operator mining equipment and labour. The base case mine production schedule involves the movement of 32 Mtpa of materials to sustain processing of 4 Mtpa of high grade ore while stockpiling up to 14 Mt of low grade ore.

The total mine life is 9.5 years for the development of a 320 m deep ultimate pit in seven stages. Pit shell and stage development is shown in Figure 1.13.1.

Figure 1.13.1 Fekola Pit Stage Design



Pit optimisations were carried out using Whittle Four-X pit optimisation software. The sequence of the pit shell obtained from optimisations were analysed to define a practical mining sequence for the pit stage designs.

1.13.2 Mine Design

Mine design is based on cutback widths of 200 to 250 m, minimum mining width of 25 m, nominal road and ramp widths of 25 m and a ramp gradient of up to 11%.

Waste dump design is based on 10 m vertical lifts with 18 degree faces and 5 m berms, with dump location considerations based on minimising haulage, surface water drainage and area availability.

1.13.3 Mine Production

The yearly mining production summary is presented in Table 1.13.1

Table 1.13.1 Mine Yearly Production Summary (Base Case)

Production Schedule 1b v3 Summary 4Mtpa Processing, 32Mtpa Mining		Total	2017 Prestrip	2018 Year 01	2019 Year 02	2020 Year 03	2021 Year 04	2022 Year 05	2023 Year 06	2024 Year 07	2025 Year 08	2026 Year 09	2027 Year 10	2028 Year 11	2029 Year 12	2030 Year 13
Physicals																
Waste Mined	kBCM	84,264	3,341	10,306	9,853	10,309	9,405	10,038	9,894	10,039	7,686	3,052	340			
Ore Mined	kBCM	17,916	283	2,365	2,373	2,131	2,259	2,092	1,801	1,784	1,185	1,298	343			
Total Mined	kBCM	102,180	3,625	12,672	12,226	12,440	11,664	12,130	11,696	11,823	8,871	4,351	683			
Waste Mined	kT	219,373	6,427	25,560	25,523	26,085	25,718	26,127	26,772	27,000	20,763	8,455	943			
Ore Mined	kT	49,173	602	6,400	6,531	5,858	6,258	5,791	4,980	4,927	3,278	3,597	949			
Au Grade	g/t	2.35	2.74	2.37	2.42	1.81	2.47	2.53	2.31	2.72	2.21	2.03	3.08			
Total Mined	kT	268,545	7,029	31,960	32,054	31,943	31,975	31,919	31,753	31,927	24,041	12,052	1,892			
	kbcm	102,180	3,625	12,672	12,226	12,440	11,664	12,130	11,696	11,823	8,871	4,351	683			
Ore Stockpiled	kT	16,439	458	3,374	2,737	2,367	2,258	1,857	1,029	1,048	631	609	71			
Stockpile Closing Balance	kT	14,105	458	3,760	6,291	8,148	10,406	12,198	13,178	14,105	13,384	12,980	9,930	5,930	1,930	0
Ore Reclaimed	kT	16,439		73	206	510		65	48	121	1,352	1,012	3,122	4,000	4,000	1,930
Total Process Feed	kT	49,173	144	3,099	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	1,930
	Au g/t	2.35	3.94	3.38	3.23	2.50	3.19	3.18	2.68	3.12	2.13	1.99	1.80	0.99	0.90	0.90
Gold Ounces																
Mill Feed	koz	3,718	18	337	415	322	410	410	345	401	274	256	231	128	116	56
Recovered	koz	3,450	17	314	387	299	382	381	320	374	253	236	213	117	106	51
Recovery	%	92.8%	93.5%	93.3%	93.2%	92.8%	93.2%	93.2%	92.9%	93.1%	92.5%	92.4%	92.3%	91.2%	91.0%	91.0%

1.13.4 Mine Operations

Mining operations estimates are based on a 3 x 8 hr shifts. Mine workforce is estimated to be 570 people, including 42 senior staff and 25 expatriate staff.

Mine startup and peak equipment requirements are detailed in Table 1.13.2.

Table 1.13.2 Operations Equipment

Mining Equipment	Start-up Qty	Peak Qty
Excavator PC2000 – Shovel	2	3
Excavator PC1250 – Backhoe	2	2
Trucks (777 D/G)	16	34
CAT D9R Dozer	3	4
CAT 844 Wheel Dozer	1	1
CAT 16M grader	2	3
Wheel Loader 988K	1	2
Wheel Loader 966K	1	2
CAT 773D WT Watertruck	2	3
CAT 330E Excavbreak	1	1
Large DHH Drills	4	5
TH Drills	2	3
RC Drills	1	2

Truck numbers increase from 16 during pre-stripping activities, up to a peak of 34 as the haul distances increase with development of deeper cutbacks. PC2000 shovels will operate on 10 m high benches with the PC1250 backhoes mainly used to mine selectively and difficult areas. Ancillary equipment numbers peak around the end of Year 2 as the mining dumping fronts advance.

An average powder factor estimate of 0.75 kg/bcm (ANFO equivalent) has been used. This is relatively high considering the strength of the fresh rock.

Although the crusher design allows for direct truck tipping, mining cost estimates assume 50% of the ROM ore will be rehandled due to variation in mine production.

1.14 Recovery Methods

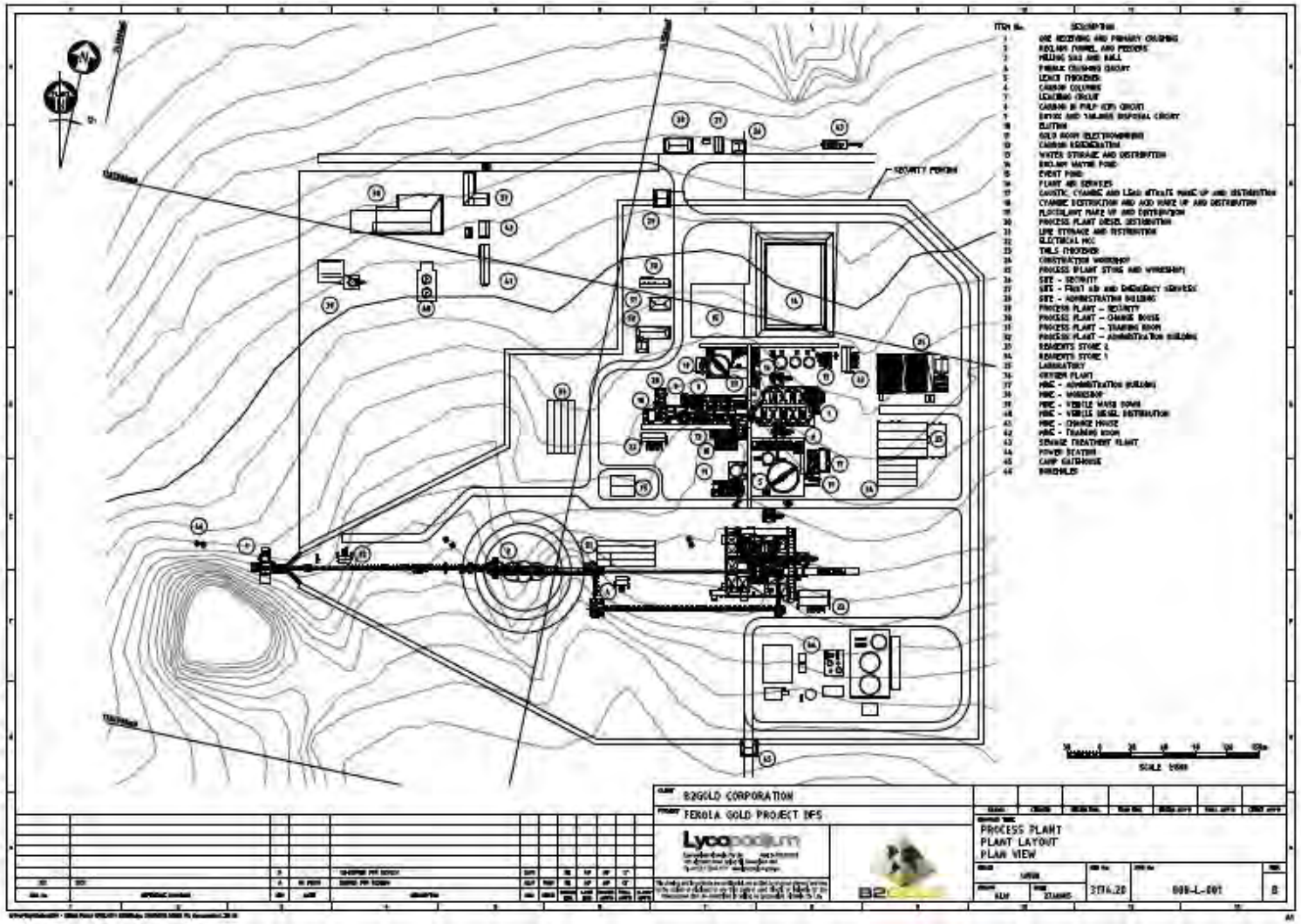
The process plant design for the Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is constructed from unit operations that are well proven in industry and consists of:

- Single stage primary crushing.
- SABC grinding circuit.
- Leach feed thickening with thickener overflow treated through a carbon in column (CIC) circuit.
- Leaching followed by carbon in pulp (CIP) adsorption.
- Elution and gold recovery to doré.

- Cyanide destruction, tailings thickening and disposal circuits.

The layout of the plant is shown in Figure 1.14.1.

Figure 1.14.1 Process Plant Layout



The processing facility is sized to treat 4 Mtpa of ore at an average head grade of 2.5 g/t Au. All major unit operations and equipment is sized with a 25% design margin to allow for future expansion capacity with minimal additional capital expenditure. The key process design criteria used as the basis of the process plant design and equipment selection are shown in Table 1.14.1.

Table 1.14.1 Summary of Key Process Design Criteria

	Units	Design	Source
Plant Throughput	tpa	4,000,000	Client
Head Grade	g Au/t	2.50	Client
Overall Gold Recovery	%	92.75	Calc from Testwork
Crushing Plant Availability	%	65.0	Client
Plant Availability	%	94.0	Client
Crushing Work Index (CWi)	kWh/t	15.8	Testwork
Bond Rod Mill Work Index (RWi)	kWh/t	21.0	Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	20.3	Testwork
SMC Axb		28.1	Consultant
Bond Abrasion Index (Ai)		0.703	Testwork
Grind Size	µm	75	Client
Leach Thickener Solids Loading	$t/m^2.h$	1.39	Testwork
CIC Superficial Upflow Velocity	m/h	75	Lycopodium
Number of Carbon Columns (Stages)		5	Client
Leach Circuit Residence Time	hrs	24	Testwork
Leach Slurry Density	% solids (w/w)	45	Client
Number of Leach Tanks		5	Lycopodium
Number of Adsorption Tanks (Stages)		6	Lycopodium
Sodium Cyanide Addition	kg/t	0.78	Testwork
Lead Nitrate Addition	kg/t	0.10	Testwork
Dissolved Oxygen Level in Leach	ppm	13 - 17	Testwork
Quicklime Addition	kg/t	0.74	Testwork
Elution Circuit Type		Zadra	Agreed
Elution Circuit Size	t	12	Agreed
Frequency of Elution	strips / week	7	Lycopodium
Cyanide Destruction Circuit Type		SO_2 & Air	Testwork
SO_2/CNwad Weight Ratio	g SO_2:g CN_{wad}	4.0	Consultant
Tailings Thickener Solids Loading	$t/m^2.h$	1.39	Testwork
Tailings Discharge Slurry Density	% solids (w/w)	60	Lycopodium

The major reagents utilized within the process plant will include:

- Quicklime (CaO) for pH control.
- Sodium Cyanide (NaCN) for gold dissolution and desorption.
- Lead Nitrate ($Pb(NO_3)_2$) for enhancing gold dissolution.

- Caustic Soda (NaOH) for carbon acid washing neutralisation and desorption.
- Hydrochloric Acid (HCl) for carbon acid washing.
- Sodium Metabisulphite (SMBS) for cyanide destruction.
- Copper Sulphate Pentahydrate ($CuSO_4.5H_2O$) for cyanide destruction.
- Flocculant for thickening.
- Antiscalant to minimise scaling in the process water distribution, reclaim water distribution, fresh water distribution, gland water distribution, and elution circuit.
- Fluxes for smelting.

The process plant will utilise process water, reclaim water, fresh water, treated water, gland water and potable water. Any shortfall of process water will be made up, preferentially, from water contained within the reclaim water pond, which includes water from the tailings decant. If insufficient water is available within the reclaim water pond, fresh water will be utilised for filling up the reclaim water pond, as required.

Air services include a compressed air system, low pressure blowers and an oxygen plant. Compressor equipment includes integral dryers so that air can be used for both plant air and instrument air duties. Low pressure blower air will be used exclusively in the cyanide destruction process and oxygen generation will be used for leaching duty.

The instrumentation and control system designed will encompass the following objectives:

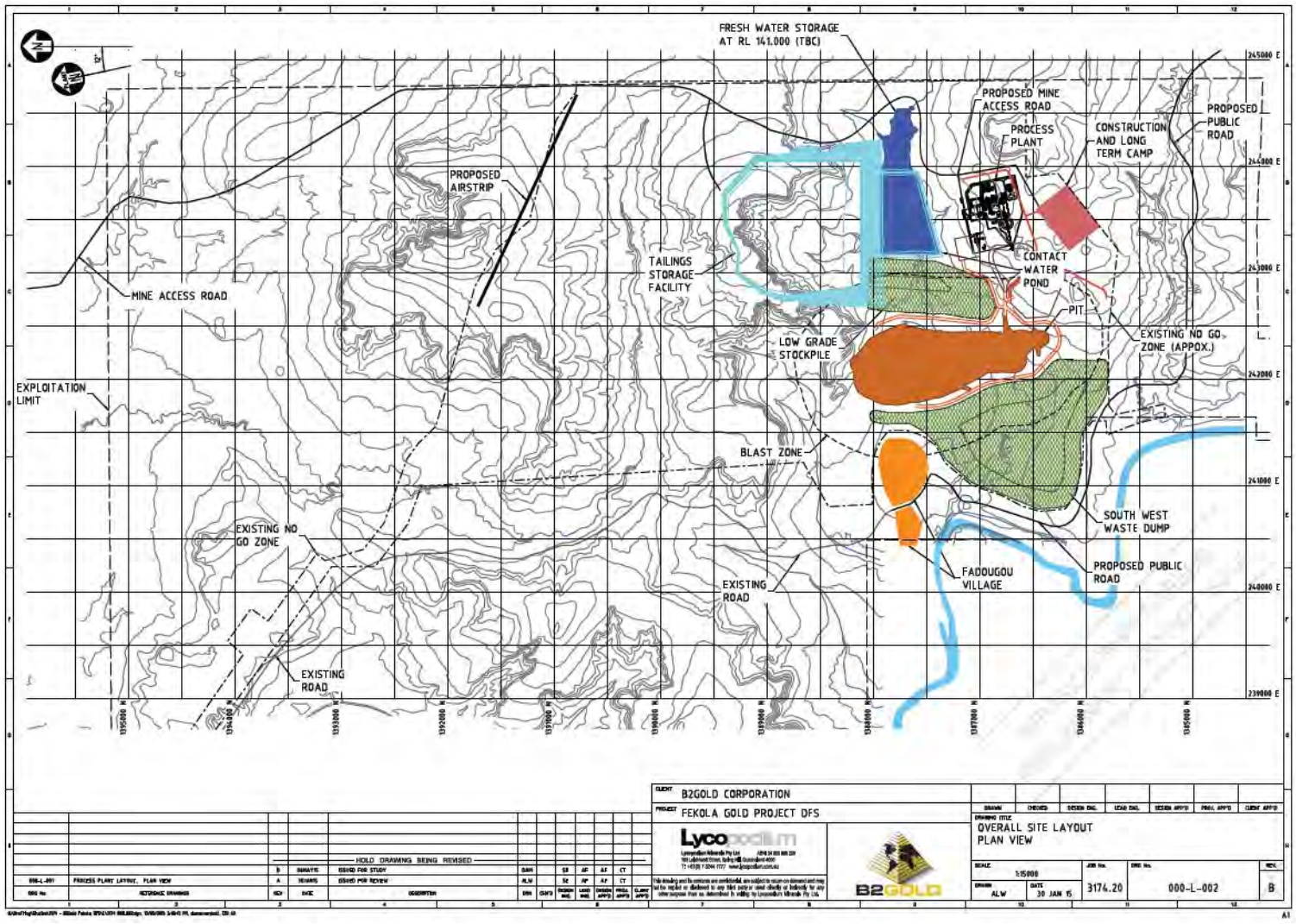
- Ease of operation: Operator intervention and remedial action will be easily accomplished.
- Interlocking: The control system will provide process interlocking between individual drives and plant areas.
- Reliability : All field equipment will be selected on its ability to operate in an environment where extreme temperatures and dusty conditions are encountered.
- Ease of maintenance: Instrumentation and control systems will be designed and installed such that maintenance requirements are minimised.

The plant control system will be designed around a moderate level of automation and monitoring.

1.15 Infrastructure

The overall site development plan is shown in Figure 1.15.1. The drawing shows the major features of the Project and its infrastructure including the process plant, tailings storage facility, accommodation camp, roads, airstrip, mine services area, mine open pit and mine waste dumps.

Figure 1.15.1 Overall Site Development Plan



1.15.2 Roads

Materials and consumables will be transported to site via a new mine access road. The new access road will link the project site with the existing Millennium Highway, 20 km from the nearby town of Kéniéba and will be a 6 m wide gravel, all weather, and free draining carriageway.

Internal plant roads will generally be 6 m wide and will be constructed flush with bulk earthworks pads.

1.15.3 Airstrip

A new 1,800 m long (1,600 m active length) x 30 m wide all weather gravel airstrip will be constructed for secure transport of bullion, transportation of construction personnel and emergency medivac purposes. The airstrip design will suit a Beech Craft 1900 type aircraft or similar.

1.15.4 Tailings Disposal (Tailings Storage Facility)

The Tailings Storage Facility (TSF) is located to the north of the process plant and pit, and adjacent to the eastern waste dump. It is designed to store a total of 62 Mt of tailings over 12 stages, with a stage lift performed every year in the dry season.

The TSF comprises of valley storage, formed by single downstream multi zoned earthfill embankment, lined with high density polyethylene (HDPE). Total footprint area is initially 69 ha at Stage 1, increasing to 200 ha for the final TSF layout.

The TSF embankments will be constructed in annual raises to suit storage requirements. Downstream raise construction methods will be utilised throughout operation. The embankment comprises an upstream low permeability zone and downstream structural fill zone. The embankment upstream face will be lined with HDPE liner.

The TSF design incorporates a basin underdrainage system to reduce pressure head acting on the geomembrane liner, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments.

Supernatant water will be removed from the TSF via submersible pumps located within decant towers, constructed at startup and raised during operation. Each decant tower will comprise a concrete base and 1.8 m diameter slotted precast concrete pipe sections. The decant tower will be surrounded by free draining coarse rockfill. The supernatant pond will be maintained on the northern edge of the TSF basin. Solution recovered from the decant system will be pumped back to the plant for reuse in the process circuit.

1.15.5 Surface Water Management

The Fekola pit footprint is located in an existing natural drainage course, with an upstream catchment of 9 km^2 requiring diversion around the pit. The site surface water management system is designed to prevent runoff from events up to and inclusive of a 1 in 100 year recurrence interval storm event from entering the pit. The surface water management system consists of the following design features:

- Diversion Structures: Designed to divert rainfall runoff from upstream catchments, around site infrastructure before discharging off site downstream of the project.
- Collection and Control Structures: Designed for containment of sediment laden runoff from the site infrastructure, via collection channels, retain coarse to fine sand particles within sediment basins on site and controlled discharge from site or storage and use within the process plant. Collection and control structures include the following infrastructure:
  - fresh Water Storage Pond (also referred to as the Clean Water Dam)
  - contact Water Storage Pond(also referred to as the Contact Water Dam)
  - various sediment control structures.

1.15.6 Falémé River Flood Assessment

An assessment of the flooding extent of the Falémé River was carried out to determine inundation risk to project infrastructure, particularly the Fekola pit. Flood modelling was completed by Knight Piésold (KP).

The predicted flood inundation level for the section of the Falémé river adjacent to the project, does not encroach on the current pit outline. However, the sediment control structure immediately downstream of the waste dump (located to the south west of the Pit) may require erosion protection on its downstream face.

Once the waste dump is in place, a peak flood level of approximately 130.5 m would be required to result in flooding. Based on the frequency analysis of the flood levels, a flood resulting in a peak flood level of 130.5 m is estimated to be greater than a 1 in 1,000 year recurrence interval.

1.15.7 Water Supply

Water for the Project will be sourced as follows:

- Pit Groundwater: Groundwater investigations estimate the average pit groundwater inflow to be 129 L/s and water quality to be pH neutral, fresh and good quality. Pit groundwater will be the main source of fresh water for process plant requirements and will be collected from dewatering bores located around the perimeter of the Fekola pit.

- Surface Water: Rainfall runoff from the upstream catchment will be collected and stored in the fresh water storage pond. Assessment of surface water samples indicates the water to be pH neutral, fresh and of good quality. Surface water collected in the fresh water storage pond will be pumped to the process plant for use as required and as a secondary water source after the pit dewatering bores.

- Falémé River Bores: Three bores have been included at the Falémé River in the event that site water quantity or quality requirements are not met as anticipated by the pit dewatering bores or surface water run-off storage.

- Potable Water Bores: Dedicated bores will be provided for potable water use at both the process plant and accommodation camp.

1.15.8 Power Supply and Distribution

Power supply to the site will be from a combination HFO and diesel fuelled power station located adjacent to the process plant. The power station will supply the main HV switchroom inside the processing plant from which power will be distributed.

The power station has been sized to accommodate a maximum demand power draw of 29.4 MW with a anticipated configuration as follows:

- 7 x 6 MW 13.8 kV medium speed generators (5 duty, 2 standby).

- 4 x 1.7 MW 13.8 kV high speed generators (emergency standby).

- 13.8 kV switchroom.

- Black start generator.

The electrical system is based on 13.8 kV distribution and 480 V, 60 Hz working voltage.

Four switchrooms are located throughout the process plant, nominally covering:

- Primary Crushing.

- Milling.

- Leach / Thickening / Water and Air Services.

- CIP / Elution / Goldroom.

Switchrooms will house 480 V motor control centres (MCCs), VVVF drives, plant control system cabinets, lighting transformers, distribution boards and UPS power distribution.

Power distribution to plant and mine infrastructure will be via kiosk substations fed from the HV distribution board.

13.8 kV overhead power lines will provide power to various remote facilities (TSF pumps, bore pumps, water storage pond pumps, etc.). Pole mounted transformers will step down the voltage at each location and supply an outdoor 480 V switchboard local to each equipment area.

1.15.9 Fuel Supply

Fuel will be trucked to site under a contract supply arrangement. Powerstation fuel storage will accommodate capacity for 30 days supply under full load conditions.

Two 1,200 m^3 diesel storage tanks are provided for fuel usage for the mobile fleet. This equates to 30 days of site diesel storage at full production.

1.15.10 Sewage and Solid Waste Management

Effluent from the process plant, mine services and administration areas will flow to a vendor package sewage treatment plant system located adjacent to the process plant. Treated effluent will be discharged to an evaporation pond. Treatment plant sludge will be suitable for direct landfill burial.

Solid wastes will be sorted and reused or recycled as far as the limited access to recycling facilities allows.

1.15.11 Accommodation Camp

The accommodation camp will be located to the south of the process plant and will provide accommodation for International expatriate and African national staff not originating from the local area.

The camp will accommodate 411 people and includes dry mess/kitchen with dining facilities for 324 people, wet mess, medical facility, admin office, mud / locker entry room, fitness centre, an indoor gymnasium with basketball court and running track, and various media, games and reading rooms.

1.15.12 Buildings and Support Facilities

Site buildings will be 'fit for purpose' industrial type structures.

Workshops, warehouses and reagent storage sheds will be constructed of a concrete slab on ground with structural steel frame and metal cladding.

Offices and amenity buildings will be prefabricated structures for ease of site installation and fit-out.

Mine Services facilities will include:

- Security gatehouse with entry boom gate control.
- Mining administration office / training room and ablutions.
- Mine changehouse.

- Mine workshop / warehouse.
- Heavy vehicle washbay facility.
- Bulk fuel storage and dispensing facilities.

Process plant facilities will include:

- High security gatehouse.
- Camp access gatehouse.
- Process plant offices.
- Plant training room.
- Plant changehouse.
- Plant ablutions.
- Lime reagent storage shed.
- Cyanide, caustic, lead nitrate and flocculant storage shed.
- SMBS, copper sulphate and acid storage shed.
- Plant laboratory.
- Plant workshop / warehouse.

Administration facilities will include:

- Low security gatehouse.
- Administration office / training room and ablutions.
- Medical / First Aid Clinic.

1.16 Market Studies and Contracts

No formal market studies have been undertaken. Fekola will produce gold doré which is readily marketable on an 'ex-works' or 'delivered' basis to a number of refineries in Europe and Africa. There are no indications of the presence of penalty elements that may impact the price or render the product unsalable.

Payment terms are widely available in the public domain and vary little from refinery to refinery. B2Gold has contracts in place for sale of gold from its four producing mines in Nicaragua, the Philippines and Namibia. It has been assumed that Fekola doré will attract similar terms.

1.17 Required Permits and Environmental Considerations

1.17.1 Environmental Consideration

An Environmental and Social Impact Assessment (ESIA) was originally completed for the Project in 2013 and approved by the Ministry of Environment and Sanitation on 29 April 2013. An update to the ESIA is currently in progress with submission planned to occur concurrently with completion of the project OFS.

As part of the ESIA a detailed assessment of potential environmental and social impacts from the development of the Project was conducted. The potential impacts were evaluated for the construction, operations, and decommissioning / post-closure phases of the Project and rated based on their significance. Potential risks associated with the proposed Project development were also assessed and discussed, as part of the impact assessment process. The assessment and discussion of potential risks associated with the Project development was broadly aligned with internationally accepted risk assessment methodologies.

An Environmental and Social Management and Monitoring Plan (ESMMP) has been developed as part of the updated ESIA, which documents the proposed mitigation and control measures to be included in the project design to meet Project regulatory requirements and reduce potential impacts from the Project to acceptable levels. The ESMMP is supported by a number of individual Management Plans (MP's) that describe how the site will meet relevant regulations, standards and guidelines and manage and minimize key environmental and social risks of the Project. The ESMMP and its supporting individual MP's are "living documents" which will be amended periodically throughout the life of the Project to reflect changes in procedures, practices and project phase.

1.17.2 Permit Consideration

The environmental permit for the Project was granted to Songhoi Resources SARL by the Ministry of Environment and Sanitation via Decision No. 2013-0033MEA-SG on 29 April 2013.

The mining permit for the Project was granted to Songhoi Resources SARL by the Secretary General via Decree No. 2014-0070/PM-RM, dated 13 February 2014.

Several additional permits and authorizations are required for the Project. These are in various stages of application, a full summary of which is provided in Section 20 of the report.

1.18 Capital and Operating Costs

1.18.1 Operating Costs

MSC prepared the mining operating cost for Fekola based on an owner operator mining strategy. Table 1.18.1 summarises the mining operating costs over the Life of Mine (LOM), based on a 4 Mtpa processing plant.

Table 1.18.1 Fekola Mine Operating Cost Summary

Cost Centre	US$ LOM	US$ /t rock	US$ /t ore	US$ /oz Au
Load Haul and Ancillary Equipment	473,035,000	1.76	9.62	137.11
Drilling and Blasting	234,490,000	0.87	4.77	67.97
Maintenance Labour and Overheads	65,968,000	0.25	1.34	19.12
Administration and Supervision	30,091,000	0.11	0.61	8.72
Total	**803,584,000**	**2.99**	**16.34**	**232.92**

Process plant and administration operating costs have been developed by Lycopodium and B2Gold based on a design treatment rate of 4 Mtpa. The operating cost estimate has been compiled from a variety of sources and is based on whole of ore treatment and a head grade of 2.5 g/t Au ore.

Table 1.18.2 Fekola Process Plant 4 Mtpa Operating cost Summary

Cost Centre	US$ M/y	US$ /t ore	US$ /oz Au
Processing Labour	3,337,124	0.83	12.10
Power	36,807,925	9.20	133.37
Consumables	29,097,026	7.27	105.43
Maintenance Materials	3,933,073	0.98	14.26
Subtotal - Processing & Maintenance	**73,184,522**	**18.29**	**265.16**
Subtotal - General & Administration	**14,893,612**	**3.72**	**53.96**
Total	**88,078,134**	**22.01**	**319.12**

1.18.2 Capital Costs

The overall OFS capital cost estimate was compiled by Lycopodium and is presented here in summary format. The capital cost estimate reflects the Project scope as described in this report. Mine capital costs (developed by MSC) and Stage 1 TSF capital cost (developed by KP) are included in the estimate represented by Table 1.18.3 below.

Table 1.18.3 Capital Estimate Summary (2Q15, ±15%)

Main Area	US$
Construction Indirects	84,200
Treatment Plant Costs	102,195,400
Reagents and Plant Service	23,326,600
Infrastructure	81,954,800
Mining	40,467,300
Engineering	13,206,300
Owners Project Costs	80,865,100
Owners Operation Costs (Working Capital)	6,963,300
Subtotal	**349,063,000**
Contingency	45,937,000
Fees, Taxes & Duties	0
Escalation	0
Grand Total	**395,000,000**

The capital cost summary excludes US$ 66.7M in mine fleet and power generation costs which are expected to be leased financed. These costs have been included in the deferred capital cost schedule.

1.18.3 Deferred Capital

Project deferred capital costs are summarised in Table 1.18.4. Deferred capital includes payments for mine fleet and power generating equipment lease financed as part on pre-production equipment purchases, increases in the mining fleet and subsequent TSF stage raises over the life of mine, and mine closure and rehabilitation costs. These costs are not included in pre production capital cost covered in Table 1.18.3 above.

Table 1.18.4 Deferred Capital Summary

Main Area	US$
Lease Payments	75,400,000
Infrastructure	51,000,000
Mining	32,900,000
Closure and Rehabilitation	20,400,000
Total	**179,700,000**

1.18.4 Project Implementation

The Project implementation plan has been developed utilising B2Gold's preferred self construct approach. Engineering consultants will provide design and procurement assistance services with B2Gold self performing construction according to the Project Schedule.

The estimated durations of key project activities are provided in Table 1.18.5.

The durations shown are preliminary and are based on receipt of the relevant authorisations to proceed.

Table 1.18.5 Key Project Milestones

Milestone Description	Planned Completion
Engineering Award Date	Week 0
Mill Contract Award	Week 1
Accommodation Camp Construction Commencement	Week 11
Concrete Construction Commencement	Week 28
Tank Construction Commencement	Week 36
SMP Construction Commencement	Week 50
E&I Construction Commencement	Week 77
Mills Mechanical Installation Commencement	Week 73
Power On	Week 105
First Gold Pour	Week 124

1.19 Economic Analysis

The project financial model was compiled using the inputs from this OFS. Key assumptions were the mining and processing production, project capital costs, and mining, processing, and general costs. The financial model is pre tax and assumes 100% ownership and a gold price of US$ 1,300/oz.

Project gold production averages 276,000 ounces of the life of the project with an average of 350,000 oz over the first seven years. The capital costs include US$ 395M in pre production capital, plus US$ 180M in sustaining capital including finance payments, tailings facility expansions, and rehabilitation and closure costs. Mining costs average US$ 2.99 over the life of the mine. Production cost is US$ 18.29/t in design conditions and averages US$ 18.30/t during the steady state production in years 2018 through 2029. General and administration costs are US$ 14.9M per year during the full mining years, decreasing to US$ 11.1M when the mining department is no longer operational.

The results of the model show robust results including a life of mine cash flow of US$ 1.66B, IRR of 35%, and an NPV of US$ 1.01B at 5%. The total cash cost including 7.65% royalties is US$ 652/oz over the life of the mine and US$ 518 over the first seven years of production (2018 to 2024).

The project has low sensitivity to capital, fuel, processing, mining, and labour costs. The project is typically sensitive to gold prices, but maintains an IRR of 33% at a US$ 1,250/ounce gold price and 19% at a US$ 1,000/oz gold price.

1.20 Adjacent Properties

There are no properties adjacent to the Fekola deposit that are relevant to the project.

1.21 Conclusions

The Project OFS has identified, defined and costed a low technical risk conventional open cut mine and carbon in pulp processing facility.

Sovereign risk is understood and mitigated by the relatively short payback of the Project. Residual environmental and social issues are low and manageable.

The Project is economically viable at current gold prices. Applying a long term gold price of US$ 1,300/oz on a flat line basis from the commencement of production, the project has an estimate Net Present Value (NPV) pre tax of US$ 1.01B at a 5% discount rate, generating a pre tax internal rate of return (IRR) of 35% and a project payback of approximately 2.25 years from first gold production. The initial mine life is estimated at 12.5 years.

Based on the Project's robust economics, B2Gold have already demonstrated a desire to move forward with project execution through the approval and commencement of early works. This includes commencement of construction of the new mine access road, preliminary site clearing, construction facility establishment and Front End Engineering Design.

1.22 Recommendations

It is recommended that B2Gold Corp advance implementation of the Project as soon as possible. Sufficient metallurgical testing and trade off studies have been completed to confidently establish the recovery process and set the design criteria. Capital and operating costs have been estimated to a level of accuracy consistent with an OFS level and no significant environmental, social or permitting issues exist, that are known of, that would prevent exploitation of the Project deposit. B2Gold's project implementation approach is a proven strategy and based on a June 2015 project approval, first gold pour can be achieved in December 2017.

FEKOLA GOLD PROJECT

NI 43-101 - INTRODUCTION

3174-STY-002

Table of Contents

			Page
2.0	**INTRODUCTION**		**2.1**
	2.1	Introduction	2.1
	2.2	Terms of Reference	2.2
	2.3	Qualified Persons	2.3
	2.4	Site Visits and Scope of Personal Inspection	2.3
	2.5	Effective Dates	2.4
		2.5.1 Information Sources and References	2.4
		2.5.2 Previous Technical Reports	2.4

TABLES

Table 2.2.1	Exchange Rates	2.3

FIGURES

Figure 2.1	Fekola Project Location	2.2

2.0 INTRODUCTION

2.1 Introduction

Qualified Persons (QPs) from B2Gold Corp. (B2Gold), Lycopodium Minerals Pty Ltd (Lycopodium), and Knight Piésold Pty Ltd (KP) have prepared a technical report (the Report) for B2Gold on the Optimized Feasibility Study (OFS) of the Fekola Gold Project (the Project) in Mali, Africa. The Project location is shown in Figure 2.1.

Following a successful acquisition of Papillon Resources Limited (Papillon) and as recommended in the previous NI 43-101 Technical Report (Fekola PEA, August 2014), B2Gold has further developed the project with geological, engineering, and technical studies including the OFS.

B2Gold currently holds 100% of Songhoi Resources Sàrl (Songhoi), the Malian holding company for the Project. A Malian exploitation company is being formed for construction and operation of the Project. Upon creation of the new Malian mining company, B2Gold's interest in the project will be 90% with a contribution of 10% share of the ownership granted to the Government of Mali. The Malian Mining Code also allows the Government of Mail to purchase for a cash value an additional 10% interest in the Exploitation Company. The Government of Mali has expressed an interest in acquiring an additional 10% interest and negotiations are ongoing. If the Government of Mali is successful, as anticipated, the final ownership of the Exploitation Company will be 80% B2Gold and 20% by the Government of Mali. B2Gold is the Project operator.

Figure 2.1 Fekola Project Location

2.2 Terms of Reference

The Report has been prepared to summarize the OFS published in June 2015, as required under NI 43-101. The Report has been prepared and is intended to be used in connection with the proposed development of the Project, and to support updated and first time disclosure by B2Gold of material scientific and technical information on a property that will be material to the shareholders of B2Gold and the stakeholders of the Project.

This information includes an updated disclosure of Mineral Resource estimates, first time disclosure of Mineral Reserve estimates, and the results of the OFS. The scope of the OFS was to assess the economic and technical viability of the Project and to provide guidance for the continued development of the Project.

All measurement units in the Report are metric. The Report was written in Canadian English. The currency is expressed in United States Dollars (US$) unless stated otherwise. Mali uses the monetary units of the Communauté Financière Africaine France (CFAF or franc, currency code XOF). For this report the exchange rates are shown in Table 2.2.1.

Table 2.2.1 Exchange Rates

	US$
AUD	0.78
ZAR	0.08
CAD	0.8
EUR	1.1
GBP	1.5
XOF	0.00165

For the purposes of this report, “B2Gold” is used as the identifier for the work programs completed on behalf of B2Gold in Mali, whether the work was performed by Papillon or Songhoi personnel.

Years and months discussed in the mine and production plan, and in the financial analysis, are presented for illustrative purposes only and are based on a project commitment decision in June 2015.

2.3 Qualified Persons

The following serve as the QPs for this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Tom Garagan, P.Geo., B2Gold. Responsible for Sections 6, 7, 8, 9, 10, 11, 12, 14, and 23.
- Peter Montano, P.E., B2Gold. Responsible for Sections 15, 16, 18, and 21.
- William (Bill) Lytle, P.E., B2Gold. Responsible for Sections 4, 5, 19, and 22.
- Ken Jones, P.E., B2Gold. Responsible for Section 20.
- Sandy Hunter, MAusIMM (CP), Principal Process Engineer, Lycopodium. Responsible for Sections 1, 2, 3, 13, 17, 18 (except 18.4, 18.5 and 18.6), 19, 21, 24, 25, 26, and 27.
- David Morgan, MIEAust CPEng, Knight Piésold. Responsible for Sections 18.4, 18.5, and 18.6.

2.4 Site Visits and Scope of Personal Inspection

The following QPs / Technical Authors have visited the site:

- Mr Tom Garagan visited the site from 2 to 4 June 2014 and 25 to 27 November 2014, during which time he inspected selected drill core and held discussions with Papillon’s on-site representatives.

- Mr Peter Montano visited the site from 5 to 7 August 2014 and again 9 to 11 December 2014, during which time he inspected the mine pit area, process plant construction area, construction aggregate sources, tailings storage facility area, and various access roads.
- Mr Ken Jones visited the site multiple times, including; 29 September to 2 October 2014, 16 to 20 February 2015, 2 to 6 March 2015, 17 to 22 April and 28 April to 7 May 2015, during these visits he inspected site facilities in relation to health, safety, environment and permitting.
- Mr Chris Orr visited the site in November 2013 and February 2014 to review the drill core from the pit geotechnical drilling and to review the pit area.
- Mr David Morgan visited the site from 11 to 12 November 2013 to guide and review hydrogeological drilling and testing and to review the TSF and other project areas.
- Mrs Sandy Hunter visited the site from 20 to 22 May 2015, during which time she inspected the process plant construction area.

2.5 Effective Dates

The report has a number of effective dates, as follows:

- Geological database closeout – 24 January 2015.
- Metallurgical information – 27 May 2015.
- Capital and operating cost information – 30 May 2015.
- Mineral tenure and surface rights – 31 March 2015.
- Social and environmental data – 31 May 2015.

The overall effective date of the Report is taken to be the date of the last material input, which is 30 June 2015.

2.5.1 Information Sources and References

Reports and documents listed in Sections 3 and 27 of this Report were used to support preparation of the Report. Additional information was provided by B2Gold and Lycopodium as needed. Supplemental information was also provided to the QPs by third party consultants retained by Papillon, Lycopodium, or B2Gold in their areas of expertise.

2.5.2 Previous Technical Reports

B2Gold filed a technical report on Preliminary Economic Assessment dated August 2014 for the Project. No other Canadian issuer has filed technical reports on the Project.

FEKOLA GOLD PROJECT

NI 43-101 - RELIANCE ON OTHER EXPERTS

3174-STY-002

Table of Contents

			Page
3.0	**RELIANCE ON OTHER EXPERTS**		**3.1**
	3.1	General Statement	3.1

3.0 RELIANCE ON OTHER EXPERTS

3.1 General Statement

The legality and currency of mining tenure and project ownership is outside the expertise of the Project team. For Item 4 on property tenure and project ownership, Lycopodium has relied entirely on the advice of tenement management experts in B2Gold through the following documents:

- Norton Rose Fulbright, 2014: Legal Report for the Médinandi Project: confidential report prepared for Papillon Resources Ltd. and B2Gold Corp, 23 June, 2014.
- Norton Rose Fulbright, 2013: Legal Report Related to Songhoi Resources Sàrl, Republic of Mali: confidential report prepared for Papillon Resources Ltd., 27 June, 2013.

Lycopodium has relied upon this advice and on advice from B2Gold that lease renewals are current. On this basis, no further advice was sought by Lycopodium on the standing of the tenements.

Lycopodium has relied upon the environmental advice from B2Gold for Item 20 of the NI 43-101 report.

Lycopodium has relied upon the financial analysis from B2Gold for Item 22 of the NI 43-101 report. Lycopodium has reviewed the inputs and basis for the financial analysis.

FEKOLA GOLD PROJECT

NI 43-101 - PROPERTY DESCRIPTION AND LOCATION

3174-STY-002

Table of Contents

			Page
4.0	**PROPERTY DESCRIPTION AND LOCATION**		**4.1**
	4.1	Overview of the Property and Title in Mali	4.1
		4.1.1 Country Overview	4.1
		4.1.2 Mineral Tenure Regime	4.1
		4.1.3 State Participation	4.3
		4.1.4 Surface Rights	4.4
		4.1.5 Royalty and Mining Tax Considerations	4.4
		4.1.6 Environmental Considerations	4.4
		4.1.7 Social Considerations	4.5
		4.1.8 Taxation Considerations	4.5
		4.1.9 Policy Perception Index	4.6
	4.2	Project Ownership	4.6
		4.2.1 Médinandi Exploitation License Ownership	4.6
		4.2.2 Songhoi Legal Standing	4.8
		4.2.3 Establishment Convention	4.8
		4.2.4 Mali Government Interest	4.9
		4.2.5 Option Agreements	4.10
		4.2.6 Legal Proceedings	4.10
	4.3	Mineral Tenure	4.10
		4.3.1 Médinandi Exploitation License	4.10
		4.3.2 Exploration Permits	4.13
	4.4	Additional Tenure in Mali	4.14
	4.5	Surface Rights	4.14
	4.6	Water Rights	4.14
	4.7	Royalties and Encumbrances	4.15
	4.8	Property Agreements	4.15
	4.9	Permits	4.15
	4.10	Environmental Liabilities	4.15
	4.11	Social License	4.15
	4.12	Comments on Section 4	4.15

TABLES

Table 4.2.1	2013 Fraser Institute Policy Perception Index Rankings, Africa	4.8
Table 4.3.1	Exploitation License Boundary Co-ordinates	4.12

FIGURES

Figure 4.3.1	Mineral Tenure Boundary, Médinandi Exploitation License	4.11
Figure 4.3.2	Exploration Permit Tenure Plan	4.13

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project, within the Médinandi Exploitation License, is located in Mali at (UTM-WGS84) 1,387,300N, 242,200E.

4.1 Overview of the Property and Title in Mali

The following summary is based on information in the prior Technical Report (B2Gold, August 2014), the public domain, CIA (2013), Deloitte (2012), Doherty (2011), the Fraser Institute (2014), and Norton Rose (2012). The QPs have not independently verified the information.

4.1.1 Country Overview

Mali is a land locked country with a total area of 1,240,000 km^2 and a total population of 12.5 million (as of July 2012). The CIA's World Fact Book (CIA, 2013) summarizes the Mali economic background as follows:

"*Among the 25 poorest countries in the world, Mali is a landlocked country highly dependent on gold mining and agricultural exports for revenue. The country's fiscal status fluctuates with gold and agricultural commodity prices and the harvest. Mali remains dependant on foreign aid. Economic activity is largely confined to the riverine area irrigated by the Niger River and about 65% of its land is desert or semi-desert.*

About 10% of the population is nomadic and about 80% of the labour force is engaged in farming and fishing. Industrial activity is concentrated on processing farm commodities."

"*The government in 2011 completed an IMF extended credit facility program that has helped the economy grow, diversify, and attract foreign investment. Mali is developing its cotton and iron ore extraction industries to diversify foreign exchange away from gold. Mali has invested in tourism but security issues are hurting the industry.*

Mali experienced economic growth of about 5% per year between 1996 and 2010, but the global recession and a military coup caused a decline in output in 2012. The interim government slashed public spending in the context of a declining state of security and declining international aid."

4.1.2 Mineral Tenure Regime

Based on information from Doherty (2011), mining activities in Mali are governed by the following legal orders and decrees:

- Order No 99-032/P-RM, of 19 August, 1999, relating to the country's Mining Code, and as modified by Order No 00-013/P-RM, of 10 February, 2000.
- Decree No 99-255/P-RM, of 15 September, 1999, pertaining to the application of the 1999 Mining Code, which was modified by Order No 00-013/P-RM, of 10 February 2000.

- Decree No 99-256/PM-RM, of 15 September, 1999 (MMM, 1999), which pertains to the approval of the model prospecting, exploration, and mining agreement to be entered into between the mineral title applicants and the State of Mali.

More recently, the following legal documents and decrees have been issued:

- Order No 2012-015 of 27 February, 2012, relating to the new Mining Code.
- Decree No 2012-311/P-RM of 21 June, 2012, pertaining to the application of the 2012 Mining Code.
- Decree No 2012-490/PM-RM of 7 September, 2012, pertaining to the approval of the model prospecting, exploration, and mining agreement to be entered into between mineral title applicants and the State of Mali.
- Decree No 2012-717/PM-RM of 20 December, 2012, pertaining to the operating and management of a fund to finance exploration, training, and promotion of mining activities.

The 2012 Mining Code and related 2012 Decrees are in force and have superseded the 1999 Mining Code and related 1999 Decrees. However, some aspects are still governed by the 1999 mining legislation for existing titles.

All new mineral titles issued after February 2012 are governed by the 2012 Mining Code and related 2012 Decrees.

The State owns all of the mineral rights and the Mines Minister has the final responsibility for the administration of mining activity, although the Minister is assisted by, and delegates certain powers to, the *Direction Nationale de la Geologies et des Mines* (DNGM).

The 1999 Mining Code (the Code, MME, 1999a) defines five types of mining titles. These titles were retained with minor modifications in the 2012 Mining Code, as follows:

- Exploration Authorization (*autorisation d'exploration*), which may be granted for three months. The maximum surface covered by this authorization is specified in the corresponding authorizing decree, and it covers specified commodities (the same area may be covered by distinct permits concerning different commodities). Such an authorization cannot be transferred to third parties by any means.
- Prospection Authorization (*autorisation de prospection*), which may be granted for three years. The tenure is renewable once with no surface reduction, assuming that the authorization holder has complied with the specifications outlined in the Code. The maximum surface covered by this authorization is specified in the corresponding authorizing decree, and it covers specified commodities (the same area may be covered by distinct permits concerning different commodities). Such an authorization can be transferred to third parties by inheritance or cession under certain conditions established by the Code. The holder should regularly deliver details of the prospection programs and reports to the Department of Mines.

- Exploration Permit (*permis de recherche*). Under the 2012 Mining Code an exploration permit is granted by order of the Minister of Mines for a period not exceeding eight years, with an initial period of two years, renewable twice, and each renewal period not exceeding two years. The 2012 Mining Code has provision for extending the second renewal for a year, if this is needed in order to complete a feasibility study. The maximum surface covered by this authorization is specified in the corresponding authorizing decree (although the maximum size of a gold permit has recently been restricted to 500 km^2), and it covers specified commodities (the same area may be covered by different permits granted for different commodities). Such an authorization can be transferred to third parties by inheritance or cession under certain conditions established by the Code. The holder should regularly deliver details of the exploration programs and reports to the Department of Mines.

- Mining License (*permis d'exploitation*), or exploitation license, which may be granted to the holder of an exploration permit or an authorization of prospection for a 30 year period, renewable for successive ten year periods until the mineral reserves within the license area are exhausted. The holder of the exploitation license under the 2012 Mining Code is required to begin exploitation within three years of the issuance of the license. The license holder must notify the Administration in charge of Mines of its intention to begin exploitation, and must mention any significant changes in key parameters of the feasibility study. If the changes impact the completion time and the viability of the proposed operation then the license holder much submit a new feasibility study. The Malian State will have a 10% carried interest that cannot be diluted by future capital contributions. The State reserves the right to increase its participation in the future by an additional 10%. Such a license can be transferred to third parties by inheritance or cession under certain conditions established by the Code.

- Small-scale Mining Authorization (*authorisation d'exploitation de petit mine*), which may be granted to the holder of an exploration permit or an authorization of prospection for a four year period, renewable for successive four year periods until the mineral reserves within the license area are exhausted. Such a permit can be transferred to third parties by inheritance or cession under certain conditions established by the Code.

Sample analysis should be conducted in Mali, unless the holder of a mineral title obtains the authorization of the Director of Mines to submit samples to another country for analysis.

Title holders must pay fixed fees for the grant, assignment, transfer, and renewal of mining titles, as well as annual surface rights. These fees are set out in the 2012 Mining Regulations.

4.1.3 State Participation

The Malian Government retains a right to a 10% non-dilutable free carried interest in the capital of a company holding an exploitation license, in addition to an option to acquire another 10% for a fair value. The 2012 Mining Code introduced an option for domestic private investors to acquire for cash at least 5% of the shares of the exploitation company, under the same conditions as other private shareholders. The conditions for the exercise of such right by Malian private investors and the exact obligations of a mining operator have not been specifically set out in either the 2012 Mining Code or the 2012 Mining Regulations.

4.1.4 Surface Rights

Mineral titles do not include any rights over the use of the soil. If the surface owner refuses the authorisation to conduct exploration or other mining activities to a permit holder then such authorisation can be legally enforced through payment of adequate compensation. If the normal land use becomes impossible due to exploration or mining activities then the surface owners could force the holder of the mineral permit to acquire the property.

For exploration permits the 2012 Mining Code required that a holder obtains consent to work the ground from local landholders, allow local communities access to communications lines, and contributes to the improvement of sanitary and scholarly infrastructure, as well as implementing recreational facilities for community and employee use.

For exploitation licenses the 2012 Mining Code requires that a holder obtains consent to work the ground from local landholders, pays for resettlement and relocation of communities if needed, contributes to the improvement of sanitary and scholarly infrastructure, as well as implementing recreational facilities for community and employee use, repair and damages arising to infrastructure from mining activities, and observe restrictions on mining activities within defined proximity limits of water wells and other infrastructure.

4.1.5 Royalty and Mining Tax Considerations

The 2012 Mining Code introduced an ad valorem tax applicable to all substances, the taxable basis of which is the square-mine value (“valeur carreau mine”) of extracted substances, exported or not, minus intermediary fees and expenses. The tax rate is based on specified Mining Groups. Gold and other precious metals would be levied at a 3% royalty rate. There is also an additional 3% tax on gold production, see Section 4.1.8.

Due to the outcome of the Traore ZTS settlement an additional 1.65% royalty is due to ZTS, see Section 4.2.1.

4.1.6 Environmental Considerations

A detailed closure plan has to be filed as part of an exploitation license application. This plan must set out how the various mining components are to be dismantled, removed, rehabilitated or regenerated. There is an expectation that the plan will contemplate progressive rehabilitation during operations. The plan may be required to be updated at five yearly intervals.

A number of provisions are included in the 2012 Mining Regulation, including the obligation to:

- Develop and implement appropriate procedures to manage chemicals and to ensure transportation, warehousing, handling and secure means of disposal of such substances as well as fuels and lubricants.
- Build onsite wastewater treatment facilities.
- Establish a program for waste reduction, sorting and recycling.

- Ensure oil and lubricants are stored appropriately, including secondary containment.
- Implement a management plan for water and mud tanks on the site.
- Formulate and implement a site specific program to monitor the quality of water collected from any pits or workings, waste rock facilities, tailings storage facilities, as well as surface water and groundwater that can be contaminated by mining activity.
- Establish regular procedures for inspection, monitoring, verification of the data recording and reporting pertaining to the tailings dams.
- Implement technical mechanisms to reduce emissions of greenhouse gas.

The 2012 Mining Code also imposes continued civil liability on the holder of an exploitation license in respect of damages or accidents caused by old equipment, even after the closure of the mine and issuance of an environmental discharge.

4.1.7 Social Considerations

Exploitation license holders must file a Community Development Plan with the exploitation license application. The plan must have been developed in consultation with the interested local communities as well as the local and regional authorities, and is required to be updated every two years.

The Mines Administration is required to create a Technical Committee on Local and Community Development. This committee is in charge of approving and implementing the Community Development Plan.

An archaeological survey is required prior to any exploitation license grant.

4.1.8 Taxation Considerations

Capital gains taxes are 10% on transfer of mining titles to third parties under the 2012 Mining Code.

A tax equal to 2% of the costs of works performed (for research permits and prospection authorizations) and 1% of the value of the project as per the feasibility study (for exploitation licenses and exploitation authorizations) is payable upon the assignment of the project to a third party.

Value added tax (VAT) is payable in Mali; however, the 2012 Mining Code has a provision that exploitation license holders have a three year VAT exemption period.

Corporate income tax is 30%. For exploitation license holders, there is a 15 year period from the start of production where the corporate income tax is reduced to 25%.

A new tax has been introduced applying to holders of an exploitation license that would produce, in one year, more than 10% of the expected quantity fixed in the annual production program approved by their shareholders' general assembly. This consists of standard taxes and rights applying to operations and results relating to overproduction.

A special tax on certain products (Impôt Spécial sur Certains Produits or ISCP), calculated on the basis of turnover exclusive of VAT, also applies and is based on the Mining Group assignment. For a gold project, the applicable ISCP rate in force upon enactment of the 2012 Mining Code was 3%.

4.1.9 Policy Perception Index

B2Gold has used the Policy Perception Index from the 2013 Fraser Institute Annual Survey of Mining Companies report (the Fraser Institute survey) as a credible source for the assessment of the overall political risk facing an exploration or mining project in Mali. Each year, the Fraser Institute sends a questionnaire to selected mining and exploration companies globally. The Fraser Institute survey is an attempt to assess how mineral endowments and public policy factors such as taxation and regulatory uncertainty affect exploration investment.

In 2013, 4,100 companies were approached and 690 companies responded. The companies participating in the survey reported exploration spending of US$ 4.6B in 2012 and US$ 3.4B in 2013.

B2Gold and the QPs have relied on the Fraser Institute survey because it is globally regarded as an independent report card style assessment to governments on how attractive their policies are from the point of view of an exploration manager or mining company, and forms a proxy for the assessment by industry of political risk in Mali from the mining perspective.

In the Fraser Institute survey, the Policy Perception Index is quantified as follows:

> *"Is a composite index, measuring the overall policy attractiveness of the 112 jurisdictions in the survey. The index is composed of survey responses to policy factors that affect investment decisions. Policy factors examined include uncertainty concerning the administration of current regulations, environmental regulations, regulatory duplication, the legal system and taxation regime, uncertainty concerning protected areas and disputed land claims, infrastructure, socioeconomic and community development conditions, trade barriers, political stability, labour regulations, quality of the geological database, security, and labor and skills availability. The Policy Perception Index is normalized to a maximum score of 100".*

Mali has a Policy Perception Index number of 0.24 in the 2013 edition of the Fraser Institute survey, as shown in Table 4.2.1. Mali is overall ranked 84th out of the 112 jurisdictions in the Fraser Institute survey.

4.2 Project Ownership

4.2.1 Médinandi Exploitation License Ownership

The Médinandi Exploitation License is held in the name of Songhoi Resources Sàrl.

Initially Songhoi's shareholders were Central African Gold plc (Central African), a company incorporated under the laws of England holding 320 shares, i.e. 80% of the shareholding, and Mani Sàrl (Mani), a company incorporated under the laws of Mali, holding 80 shares, i.e. 20% of the shareholding. Their corporate relations were governed by a joint venture agreement dated 25 July 2006 which was amended on 15 September 2006 (the Joint Venture Agreement).

On 18 December 2009, Central African and Papillon Resources Limited (Papillon), a company incorporated under the laws of Australia (and previously known as Colonial Resources Limited) entered into a sale of shares agreement pursuant to which Central African assigned its 320 shares in Songhoi to Papillon (the First Share Sale Agreement), as further witnessed by a deed of assignment dated 10 March 2010 (the Deed of Assignment).

Papillon and Songhoi subsequently entered into a sale of shares agreement pursuant to which Mani assigned 40 of its shares in Songhoi to Papillon (the Second Share Sale Agreement). In addition, during an extraordinary general assembly held on 15 August 2012, Papillon and Mani, being the sole shareholders of Songhoi, decided to amend the articles of association accordingly.

In October 2014 B2Gold and Papillon completed a merger agreement under the laws of Australia and Canada with the resulting company retaining the name of B2Gold and acting as the Project operator. This merger included all Papillon assets, including 90% of Songhoi.

In January 2015 B2Gold purchased Mani's 10% interest in Songhoi for a package consisting of cash payments, B2Gold stock payments, a 1.65% operating royalty, and an agreement to enter into an exclusive fuel supply agreement for the Project with ZTS. In connection with the purchase all existing ZTS legal claims were terminated.

B2Gold is now the 100% owner of Songhoi and the Project.

Table 4.2.1 2013 Fraser Institute Policy Perception Index Rankings, Africa

		Score					Rank				
		2013	2012/ 2013	2011/ 2012	2010/ 2011	2009/ 2010	2013	2012/ 2013	2011/ 2012	2010/ 2011	2009/ 2010
Africa	Angola	0.10	*	*	*	*	109/112	*	*	*	*
	Botswana	0.62	0.60	0.75	0.68	0.68	14/112	16/96	1/93	7/79	7/72
	Burkina Faso	0.48	0.55	0.63	0.71	0.70	33/112	27/96	13/93	6/79	4/72
	DRC (Congo)	0.28	0.23	0.38	0.21	0.30	74/112	83/96	55/93	70/79	56/72
	Eritrea	0.33	*	*	*	*	60/112	*	*	*	*
	Ethiopia	0.46	*	*	*	*	38/112	*	*	*	*
	Ghana	0.45	0.56	0.60	0.57	0.60	40/112	23/96	17/93	24/79	18/72
	Guinea (Conakry)	0.27	0.29	0.63	0.63	*	77/112	74/96	58/93	56/79	*
	Ivory Coast	0.21	*	*	*	*	91/112	*	*	*	*
	Kenya	0.35	*	*	*	*	58/112	*	*	*	*
	Liberia	0.29	*	*	*	*	72/112	*	*	*	*
	Madagascar	0.20	0.12	0.38	0.41	*	95/112	90/96	52/93	46/79	*
	Mali	0.24	0.33	0.55	0.59	0.64	84/112	65/96	26/93	21/79	10/72
	Mozambique	0.25	*	*	*	*	79/112		*	*	*
	Namibia	0.45	0.50	0.45	0.55	0.58	39/112	35/96	44/93	29/79	24/72
	Niger	0.13	0.40	0.38	0.42	*	108/112	52/96	52/93	44/79	*
	Nigeria	0.18	*	*	*	*	97/112	*	*	*	*
	Sierra Leone	0.25	*	*	*	*	80/112	*	*	*	*
	South Africa	0.37	0.28	033	0.28	0.39	55/112	77/96	62/93	66/79	45/72
	Tanzania	0.35	0.42	0.55	0.58	0.47	57/112	47/96	25/93	23/79	35/72
	Zambia	0.47	0.38	0.47	0.46	0.53	36/112	58/96	39/93	37/79	28/72
	Zimbabwe	0.15	0.10	0.21	0.16	0.21	103/112	92/96	87/93	74/79	67/72

Table sourced from the 2013 Fraser Institute Annual Survey of Mining Companies report.

4.2.2 Songhoi Legal Standing

Legal opinion was provided that supports that Songhoi is a limited liability company (société à responsabilité limitée or Sàrl) that is duly incorporated with the Trade and Property Credit Register (Registre du Commerce et du Crédit Mobilier or RCCM) and validly exists under the laws of Mali. The company's purpose is to conduct exploration and mining activities in Mali and overseas. The company is managed by a managing director (gérant).

4.2.3 Establishment Convention

The Médinandi Exploration Permit was initially granted to Zoumana Traore Sàrl (ZTS) by Order No. 041113 dated 27 May 2004 for the exploration of gold and other mineral substances within Mineral Group II.

The Médinandi Exploration Permit was assigned twice, as follows:

- To the benefit of Mani by Order No. 06-1413 dated 3 July 2006 (the First Assignment Decree) authorising the assignment to Mani, as a result of the permit assignment agreement dated 2 May 2006 (the First Assignment Agreement).

- To the benefit of Songhoi by Order No. 06-2761 dated 13 November 2006 (the Second Assignment Decree) authorising the assignment to Songhoi.

ZTS and the Government of Mali entered into a corresponding establishment convention on 25 February 2004 pertaining to the Médinandi Exploration Permit (the Establishment Convention), in accordance with the 1999 Mining Code and the Establishment Convention Model.

None of the 1999 Mining Code, the 2012 Mining Code, or the various agreements, including the joint venture agreement, specifically provided for an assignment of the rights and obligations of ZTS under the Establishment Convention, firstly to Mani and subsequently to Songhoi. The QP was provided with legal opinion to the effect that the initial Establishment Convention would automatically be transferred upon transfer of the Médinandi Exploitation License. Further legal actions have resulted in a settlement with ZTS, refer to Section 4.2.1.

The Establishment Convention provides that the Government must, within 90 days from the date of submission of the application for grant of the exploitation license, express in writing its intention of acquiring cash participation and the percentage of such participation in the capital of the registered Malian entity that holds the exploitation license.

The Establishment Convention will expire on 25 February 2034, which is before the expiry date of the Médinandi Exploitation License. A new establishment convention or an extension of the Establishment Convention will thus need to be negotiated for the remainder term of the Médinandi Exploitation License.

4.2.4 Mali Government Interest

Under the 2012 Mining Code, upon the grant of an exploitation permit the holder of an exploitation license must take steps to create an exploitation company incorporated under the laws of Mali. This requirement is restated in the Establishment Convention, and in the joint venture agreement. B2Gold is currently in the process of creating the new Malian exploitation company.

B2Gold has received notice of participation from the government of Mali (refer to Section 4.1.3). This will include the 10% non-dilutable free carried interest, plus up to an additional 10% for cash value. Following the Government of Mali's participation the new Malian company shareholding will be distributed, as follows:

- 80 to 90% for B2Gold.
- 10 to 20% for the Government of Mali.

It is not certain whether the Project will fall under the 2012 Mining Code, or whether the Project would be considered to be grandfathered under the previous 1999 Mining Code. This will be determined when the new Malian operating company is established and the convention terms are finalized. However, B2Gold is operating under the assumption that the 2012 Mining Code will apply.

4.2.5 Option Agreements

Under the First Permit Assignment Agreement, ZTS was granted an option to acquire 2.5% of the shares in the new Malian company. In this respect, the terms of the joint venture agreement entail that this option is to be exercised over the shares that Papillon would hold in the new Malian company. The joint venture agreement further provides that the option is to be exercised at a value to be set by an expert approved by the parties.

Under the joint venture agreement entered into between Central African (being Papillon's predecessor in title) and Mali Mining House SA (Mali Mining) on 9 December 2005, and as amended firstly on 25 July 2006 and subsequently on 12 December 2006 (the MMH Joint Venture Agreement), Central African granted to Mali Mining and/or Mali Mining's shareholders an option to acquire up to 3% maximum of the shares in the new Malian company at a price set by an expert approved by the parties. Pursuant to the MMH Joint Venture Agreement, Central African agreed to pay to MMH 1% of any net (after tax) dividend that it would receive from the new Malian company whether this dividend results from the exercise of mining activities or the sale of a mining title.

Following resolution of the ZTS proceedings (see Section 4.2.1) there are no current option agreements.

4.2.6 Legal Proceedings

Papillon was involved in legal proceedings with ZTS prior to and during the B2Gold / Papillon merger. Under a judgement from the Commercial Court of Bamako, ZTS was considered to hold 17% of Songhoi's share capital, 10% of which is already indirectly held by ZTS through Papillon's joint venture partner, Mani Sàrl. Papillon had appealed this judgement.

Papillon had also initiated arbitral proceedings in Paris through the International Chamber of Commerce (ICC) in order to secure its rights against ZTS and other respondents. This action had been registered by the ICC Secretariat.

Refer to Section 4.2.1 for the conclusion and closure of the legal proceedings.

4.3 Mineral Tenure

4.3.1 Médinandi Exploitation License

A mining lease (the Médinandi Exploitation License) was granted to Songhoi for the Project on 13 February 2014, under permit number 0070/PM-RM. The requirements to maintain the license include construction of the mine as planned, payment of nominal annual fees and filing of various standard reports.

The exploitation license covers the entire area of the former Médinandi Exploration Permit and is 75 km^2 in areal extent. The exploitation license is valid for a 30 year term, expiring February 2044.

The exploitation license boundary outline and boundary apex co-ordinates are given in Figure 4.3.1 and Table 4.3.1 respectively.

Figure 4.3.1 Mineral Tenure Boundary, Médinandi Exploitation License

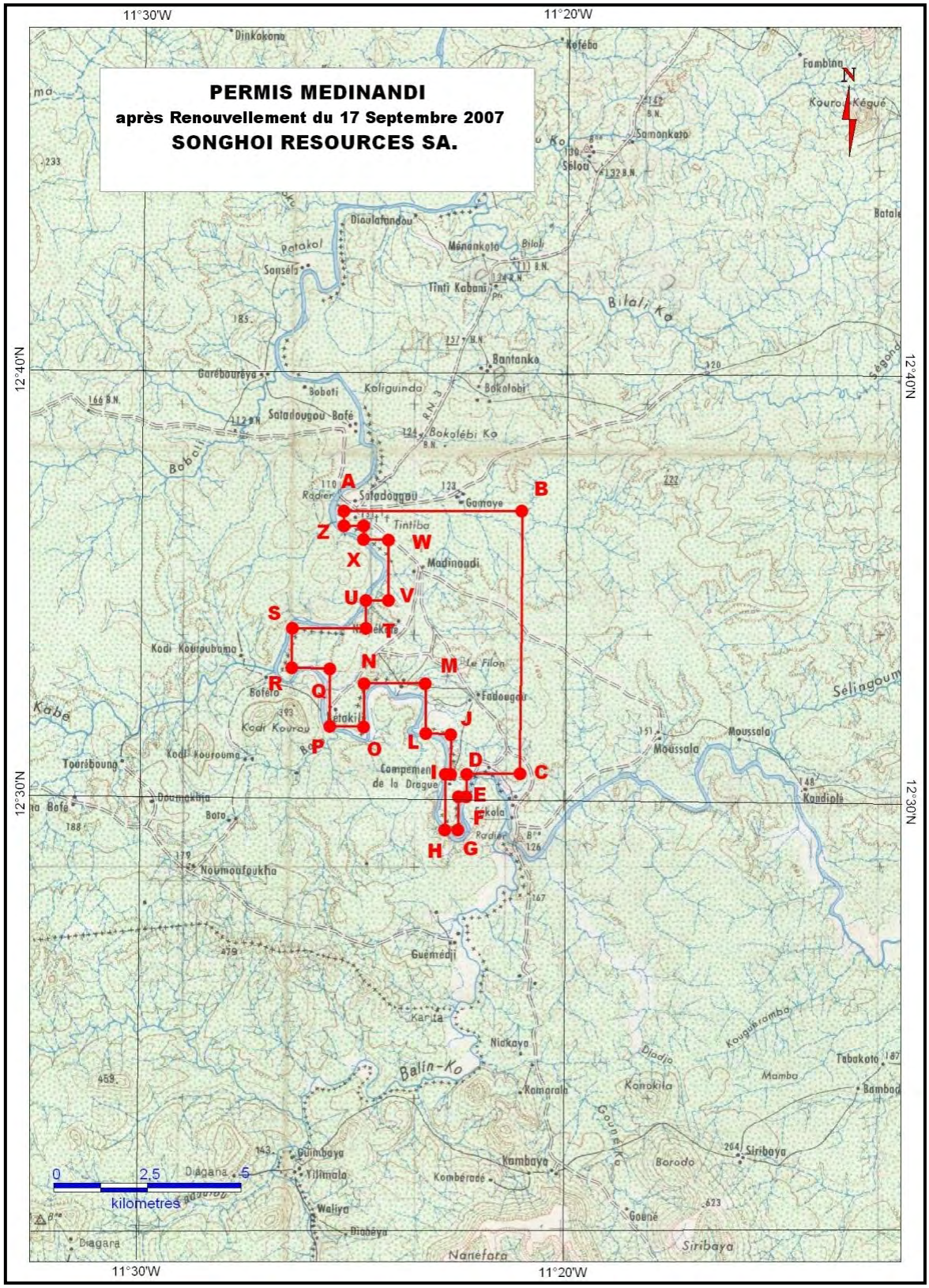


Table 4.3.1 Exploitation License Boundary Co-ordinates

Point	Longitude	Latitude
A	11° 25' 16"	12° 36' 44"
B	11° 21' 03"	12° 36' 44"
C	11° 21' 03"	12° 30' 34"
D	11° 22' 19"	12° 30' 34"
E	11° 22' 19"	12° 30' 03"
F	11° 22' 30"	12° 30' 03"
G	11° 22' 30"	12° 29' 15"
H	11° 22' 49"	12° 29' 15"
I	11° 22' 49"	12° 30' 34"
J	11° 22' 42"	12° 30' 34"
K	11° 22' 42"	12° 31' 29"
L	11° 23' 18"	12° 31' 29"
M	11° 23' 18"	12° 32' 40"
N	11° 24' 46"	12° 32' 40"
O	11° 24' 46"	12° 31' 39"
P	11° 25' 35"	12° 31' 39"
Q	11° 25' 35"	12° 33' 00"
R	11° 26' 29"	12° 33' 00"
S	11° 26' 29"	12° 33' 57"
T	11° 24' 44"	12° 33' 57"
U	11° 24' 44"	12° 34' 37"
V	11° 24' 13"	12° 34' 37"
W	11° 24' 13"	12° 36' 02"
X	11° 24' 49"	12° 36' 02"
Y	11° 24' 49"	12° 36' 21"
Z	11° 25' 16"	12° 36' 21"

Figure 4.3.2 Exploration Permit Tenure Plan

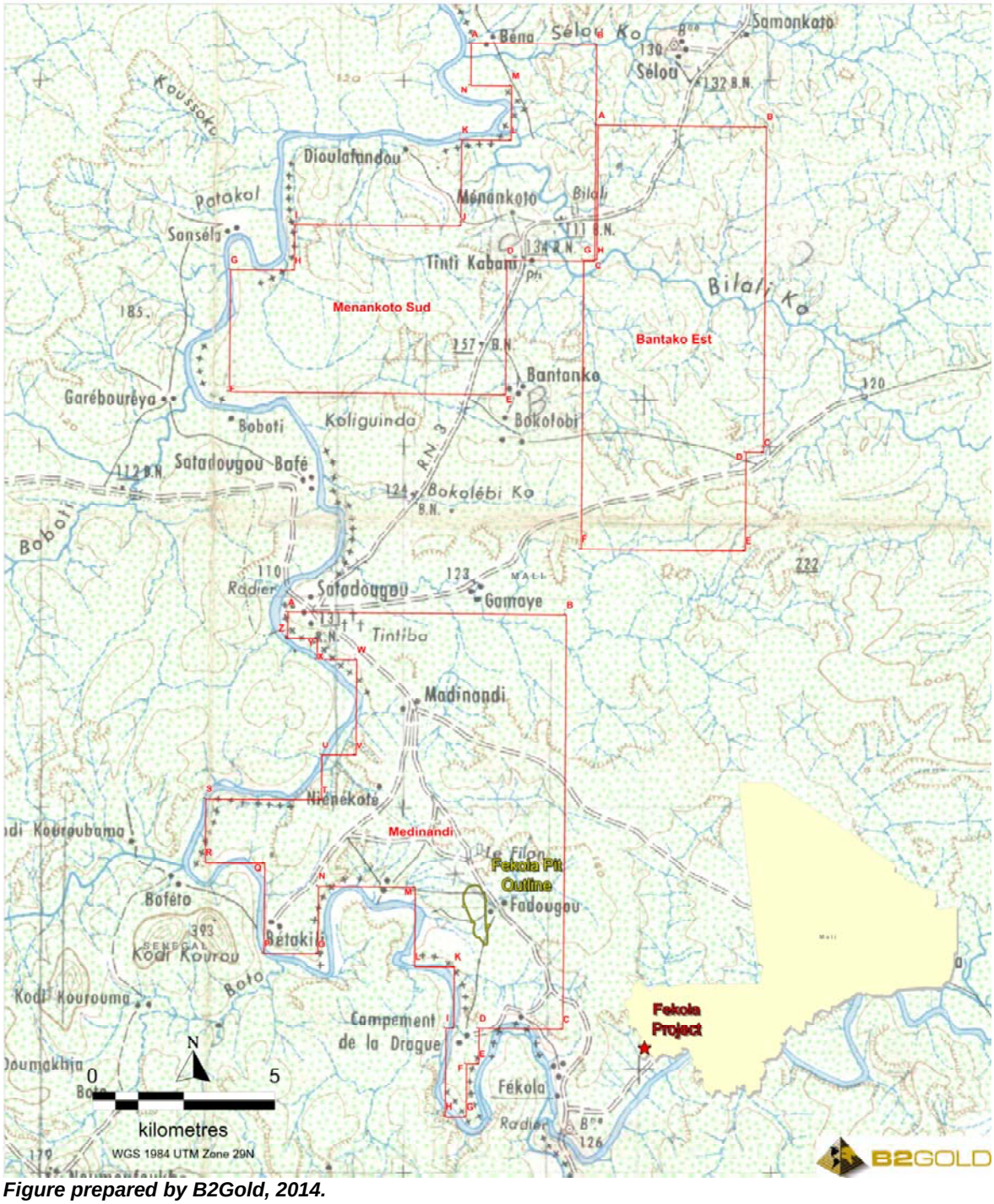


Figure prepared by B2Gold, 2014.

4.3.2 Exploration Permits

Two exploration permits, Menankoto Sud and Bantako Est, held by Songhoi are within close proximity to the Médinandi tenement (Figure 4.3.2).

The Menankoto Sud exploration permit is 52 km^2 in area and is located approximately 13 km to the north of the Médinandi Exploitation License. The permit was granted on 4 February 2014, and is valid until 4 February 2017, with two, two year renewals possible. B2Gold has a 95% interest in Menankoto Sud. Minimum expenditures are required to maintain the permit, and comprise CFAF 65.4M in Year 1, CFAF 210.5M in Year 2, and CFAF 287M in Year 3, for a total required expenditure commitment of CFAF 562.9M.

The Bantako Est exploration permit is 57 km^2 in area and is located less than 5 km from the northeast corner of the Médinandi Exploitation License. B2Gold has a 90% interest in Bantako Est. The permit was granted on 30 December 2011, and expired on 30 December 2014. The renewal was submitted as required and is currently awaiting Mali government signatures. Minimum expenditures are required to maintain the permit, and comprise CFAF 150M in Year 1, CFAF 205M in Year 2, and CFAF 335M in Year 3, for a total required expenditure commitment of CFAF 690M.

4.4 Additional Tenure in Mali

B2Gold has additional tenure holdings in Mali which are at a grassroots exploration stage. These tenements are not considered to be part of the Project as defined due to their distance from the Médinandi Exploitation License; the distances being such that there is no likelihood of shared infrastructure with the Project.

4.5 Surface Rights

Malian law provides for private individuals and companies to own surface rights under a formal titling and registration system, but in the Project area there are no private surface owners. The subsistence farmers typically use the land under a customary use and lineage system where no formal title has been registered but the land is allowed to be used. This recognition is only tacit and the Malian government can appropriate the land as it sees fit.

All of the surface rights in the Project area are under the ownership of the State of Mali and have not been registered to any private entity.

There are a number of small villages in the license area, but there are currently no inhabitants in a “no go” zone, which is the area required for mining operations, infrastructure, and a 500 m buffer zone around the active blasting area. Previous farmers and other inhabitants have been relocated and compensation has been paid.

There are no future payments or liabilities associated with the relocation effort and there are currently no inhabitants remaining in the no go zone. B2Gold is reviewing a larger scale relocation effort for the nearby village (Fadougou), but this is not a required component of the Project and does not represent a hindrance to the Project development.

4.6 Water Rights

The primary process water source in the ESIS and the PEA was assumed to be the Falémé River. This would require a permit from the OMVS which is the regulatory body which manages the transnational boundary. The water study in the ESIS would be the basis for obtaining this permit.

Subsequent investigations have determined that water from dewatering holes for the open pit area will provide sufficient process water. This concept will provide cleaner, higher quality water and will avoid any dry season Falémé River pumping problems. A hydrological program, including pump tests, has been initiated and will refine the site wide water balance and water extraction quantities.

The Project does not currently have any water permits. All permits required for the extraction and use of water will be obtained once water requirements and water sources have been better defined.

4.7 Royalties and Encumbrances

As noted in Section 4.1.5, a 3% royalty (mining tax) is payable to the Malian government. There is an additional 3% tax on the sales of gold (ISCP, see Section 4.1.8). The total Malian royalty payments on production will be 6%. As noted in Section 4.2.1 an additional royalty of 1.65% is payable to ZTS. The total royalties payable are therefore 7.65%.

4.8 Property Agreements

The Project is subject to the ownership and joint venture agreements discussed in Section 4.2.

4.9 Permits

Project permit requirements are discussed in Section 20.

4.10 Environmental Liabilities

The current environmental liabilities for the Project are discussed in Section 20.

4.11 Social License

Considerations of stakeholder interests and social license for current and planned operations are discussed in Section 20.

4.12 Comments on Section 4

In the opinion of the QP:

- B2Gold currently holds a 100% interest in the Project. On creation of the new Malian mining company, B2Gold's interest will be 90% with the contribution of a 10% share of the ownership to the Government of Mali. The Malian Mining Code also allows the Government of Mail to purchase (at market terms) an additional 10% interest in the Exploitation Company. The Government of Mali has expressed an interest in acquiring an additional 10% interest and negotiations are ongoing. If the Government of Mali is successful, as anticipated, the final ownership of the Exploitation Company will be 80% B2Gold and 20% by the Government of Mali

- B2Gold is the current Project operator.

- Legal opinion provided supports that the Médinandi Exploitation License is valid and mineral tenure is sufficient to support declaration of Mineral Resources and Reserves.
- Royalties in the form of the Malian mining tax will be payable.
- The QP is not aware of any significant environmental, social or permitting issues that would prevent future exploitation of the Project deposits other than as discussed in this Report.

FEKOLA GOLD PROJECT

NI 43-101 - ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

3174-STY-002

Table of Contents

			Page
5.0	**ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY**		**5.1**
	5.1	Accessibility	5.1
	5.2	Climate	5.1
	5.3	Local Resources and Infrastructure	5.1
	5.4	Physiography, Topography, Elevation and Vegetation	5.2
	5.5	Comments on Section 5	5.3

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located within the Kayes Region in south-western Mali, on the border of Mali and Senegal. It is approximately 210 km south of Kayes and approximately 40 km south of the city of Kéniéba.

Access to the Project is either from Dakar or from Bamako by road. From Bamako it is approximately 480 km along the recently completed Millennium Highway from Bamako to Kéniéba, then a further 45 km on gravel / dirt roads from Kéniéba to the project site. A new road from site to the Millennium Highway is currently under construction, and will allow all-weather two-way traffic for construction and operating traffic. The new road will generally consist of upgraded existing roads and will intersect with the Millennium Highway approximately 2.5 km north of the village of Segondo.

Current air access is via charter flights to an airstrip 1 km east of Kéniéba, then by road to site. A Fekola site airstrip is currently in construction. The airstrip will be a gravel / dirt surface at least 1,600 min length.

5.2 Climate

The Project area is in the pre-Guinean climate zone, where the average annual rainfall is greater than 1,000 mm and may reach 1,200 mm with a maximum of 85 days of rain and a rainy season lasting three months (July through September), characterized by torrential rains. Average annual rainfall is highly variable.

The temperatures vary from a maximum 46ºC and minimum 10°C depending upon season, with a mean annual temperature of 28°C.

Wind speeds are generally calm, normally varying between 1 m/s and 5 m/s. The operating design wind speed is 25 m/s and the maximum wind speed is 45 m/s. The predominant wind direction is from the north-west, but this varies seasonally and day versus night.

Exploration activities are minimal during the period from July to September due to the rains. Construction, mining, and processing activities will be conducted year-round.

5.3 Local Resources and Infrastructure

Due to the remote location the Project has limited existing infrastructure. Proposed project infrastructure including sources of water, power, communications, mining areas, processing areas, waste disposal areas, and tailings disposal areas are described in Section 18 of this Report.

5.4 Physiography, Topography, Elevation and Vegetation

The site is characterized by various laterite plateaus that rise approximately 30 to 40m above the surrounding landscape. These plateaus are relatively flat and generally drain in a westerly direction. At the edge of the laterite plateaus the topography is steep in comparison to the general topography of the site. Elevation ranges from approximately 130 to 190 m above sea level.

A number of drainage lines cross the property and drain from east to west. The Falémé River is adjacent to the Project site and flows in a northerly direction with meanders to the west and south of the site.

The Project is located in the Guinean bioclimatic zone which corresponds to high humidity and ecosystems of open forests. Vegetation types observed in the Project area include:

- Wetlands on laterite hardpans: this is a much localized vegetation association that is confined to depressions in laterite hardpans that hold water through the wet season. The wetlands are usually not more than 500 m^2 in area. Open water is usually limited, and much of the surface area is usually covered in hydrophytic vegetation.

- Grassland on laterite hardpans: this is a widespread vegetation association that occurs on all the laterite hardpans, which are characterized by very shallow to non-existent soils and many areas of exposed sheetrock. Grasses are usually not higher than 200 mm, while scattered woody shrubs can be up to 2 m in height. Grass canopy cover varies from 50 to 80%.

- Riparian forest: most prominent along the perennial Falémé River, and is most developed on islands in the river.

- Thicket on laterite scarps: a complex of five vegetation associations that are all strongly associated with the rocky edges of laterite scarps and the steep slopes below these scarps.

- Woodland on alluvium: a tall semi-deciduous closed woodland or forest community is confined to alluvial soils along valley bottoms.

- Woodland on silt: occurs mostly in valley bottoms and foot-slopes.

- Woodland on ferrocrete: found across all landform types (from crests to valley bottoms), while occurring mainly on mid-slopes, upper slopes, and crests.

- Woodland on steep sandy slopes: a much localized vegetation association which appears to be confined to a small area of steep, north facing slopes in the north-eastern quarter of the Project area.

Additional information on the Project flora is included in Section 20.

5.5 Comments on Section 5

In the opinion of the QP:

- There is sufficient suitable land available within the exploitation concession and more specifically within the planned project area for the planned tailings disposal, waste rock dumps, open pit, mining infrastructure, process plant, workshops, offices, power plant, and other planned facilities.
- The mining license provides the license holder with exclusive access to and use of the Project area. This does not give the license holder ownership of the land, but does make the land available for construction, operation, and other infrastructure needs.
- No surface rights are currently held. However, the process for obtaining such rights is well understood and such rights will be granted with formation of the exploitation company.
- Construction and mining activities are reasonably expected to be conducted year round.

FEKOLA GOLD PROJECT

NI 43-101 - HISTORY

3174-STY-002

Table of Contents

			Page
6.0	**HISTORY**		**6.1**
	6.1	Ownership History	6.1
		6.1.1 Legacy Period (1954 to 2001)	6.1
		6.1.2 Recent Period (2006 to Present)	6.1
	6.2	Exploration History	6.2
		6.2.1 Legacy Period	6.2
		6.2.2 Recent Period	6.3
	6.3	Resource History	6.6

TABLES

Table 6.2.1	Summary Table of Historical Activities, Médinandi Exploitation Permit	6.4
Table 6.3.1	January 2013 Papillon Historical Resource Estimate	6.8
Table 6.3.2	August 2013 Papillon Historical Resource Estimate	6.9

FIGURES

Figure 6.2.1	Prospect and Deposit Location Plan, Médinandi Exploitation Permit	6.6

6.0 HISTORY

6.1 Ownership History

6.1.1 Legacy Period (1954 to 2001)

The early ownership history of the Médinandi exploration permit area in its various forms has largely been gleaned from the prospectus filed with the ASX by Colonial Resources dated 25 January 2010.

The earliest reference to exploration activity in the area relates to work done by an unnamed French prospector who discovered gold in quartz veins at the Fadougou Main Zone in 1953. This activity is recorded in two prospecting reports prepared in 1954 and 1955. These reports have not been found by B2Gold.

Société Nationale de Recherches et d'Exploitation des Ressources Minières de Mali (Sonarem) commenced reconnaissance activities in 1962, funded by Russian interests from 1966 to 1970. This continued from 1975 to 1984 with input from the Bureau de Recherches Géologiques et Minières (BRGM), the French Geological Survey. Guefest Company (Guefest), a Russian mining consortium co-operated with Sonarem from 1992 to 1996.

Western African Gold and Exploration S.A. (WAG) were assigned the Guefest interest in the Médinandi area in 1996 following an agreement between Guefest, Eagle River International Limited (Eagle River), Azco Mining Inc (Azco) and Lion Mining Finance Limited (Lion Mining).

Randgold Resources Ltd. (Randgold) entered into a joint venture (JV) agreement with WAG / Azco on the project area in 1998. Work completed by the JV is recorded to 2001.

6.1.2 Recent Period (2006 to Present)

Central African Gold (Central African) acquired an interest in the project in 2006. Central African's interest was transferred to Colonial Resources Limited (subsequently Papillon) during 2010. The Mali registered holding company Songhoi Resources SARL (Songhoi) held the exploration permits and eventually the Médinandi Exploitation License, which contains the Project.

Initially Songhoi's shareholders were Central African Gold plc (Central African), a company incorporated under the laws of England, holding 320 shares, i.e. 80% of the shareholding, and Mani Sàrl (Mani), a company incorporated under the laws of Mali, holding 80 shares, i.e. 20% of the shareholding. Their corporate relations were governed by a JV agreement dated 25 July 2006, which was amended on 15 September 2006 (the JV Agreement).

On 18 December 2009, Central African and Papillon Resources Limited (Papillon), a company incorporated under the laws of Australia (and previously known as Colonial Resources Limited) entered into a sale of shares agreement pursuant to which Central African assigned its 320 shares in Songhoi to Papillon (the First Share Sale Agreement), as further witnessed by a deed of assignment dated 10 March 2010 (the Deed of Assignment).

Papillon and Songhoi subsequently entered into a sale of shares agreement pursuant to which Mani assigned 40 of its shares in Songhoi to Papillon (the Second Share Sale Agreement). In addition, during an extraordinary general assembly held on 15 August 2012, Papillon and Mani, being the sole shareholders of Songhoi, decided to amend the articles of association accordingly.

Following this amendment the ownership of Songhoi was Papillon holding 360 shares, representing 90% of Songhoi's shareholding and Mani holding 40 shares, representing 10% of Songhoi's shareholding.

On 3 October 2014, B2Gold Corp. acquired 100% of the ordinary shares of Papillon Resources Limited, now referred to as Papillon Resources Pty Ltd. ("Papillon") by way of an Australian scheme of arrangement. The scheme of arrangement was carried out pursuant to the terms and conditions contained in a merger implementation agreement (the "Papillon Merger Agreement") dated 3 June 2014 between B2Gold and Papillon. On completion of the transaction, Papillon became a wholly-owned subsidiary of B2Gold Corp.

In January 2015, B2Gold purchased the remaining 10% non-controlling interest in the Project from Mani, the minority shareholder of Songhoi. Following this purchase Songhoi is fully owned by B2Gold Corp.

Negotiation of the Malian convention related to the formation of an Exploitation company (required for operation of a producing mine) is underway and will include government participation as described in Section 4.1.3 of this report.

6.2 Exploration History

6.2.1 Legacy Period

Quartz veining in the Médinandi general area was first identified in 1953 by a French prospector at a prospect known as the Fadougou Main Zone. A gold bearing quartz vein with a strike extent of about 280 m and widths varying from 0.2 to 1 m were reported.

Sonarem commenced reconnaissance activities in 1962 and following funding by Russian interests between 1966 and 1970, undertook more detailed exploration in the Fadougou Main Zone area. Additional work was performed by Sonarem in cooperation with BRGM between 1975 and 1984. Work undertaken in collaboration with Guefest between 1992 and 1996 appears to have included geological mapping, reconnaissance trenching and surface sampling, geophysical surveys, auger and core drilling. There is very limited documentation available, other than through secondary sources of this work, and none of the drilling is included in the current B2Gold database. BRGM is reported to have undertaken a resource estimate at Fadougou Main Zone (Colonial Resources, 2010).

During the WAG/Azco period between 1992 and 1996 work included regional mapping, ground induced polarization (IP) geophysical surveys, soil and termite geochemical sampling, trenching, auger and reverse circulation (RC) drilling, and resource estimates for the Fadougou Main Zone area, which was subsequently referred to as the Fadougou deposit, and more recently referred to as the Fekola Satellites.

Between 1998 and 2001, the Randgold and WAG/Azco JV undertook interpretation of Landsat and aeromagnetic data, geological and regolith mapping, regional geochemical soil and rock, compilation of data from previous work, and updated mineral resource estimate for the Fadougou deposit.

This early work from 1955 to 2001, referred to as legacy data in this Report, identified a number of prospects and targets in the permit area, in addition to the Fadougou Main Zone; these included Fadougou Southeast, Fadougou South Southeast, Fadougou Northeast, Tintiba, Médinandi, and Betakili targets.

6.2.2 Recent Period

From 2006 to 2008, Central African (Papillon) completed mapping, soil geochemical surveys, ground IP and airborne magnetic and electromagnetic (EM) surveys over the project area, together with RC and core drilling (130 holes) over the Médinandi and Fadougou zones.

Fadougou deposit resource estimates (and the estimate referred to by Papillon as the Fekola Satellites) were prepared in 2009, 2011, and 2014. These estimates are not considered current by B2Gold.

On acquisition by Papillon, a large package of soil samples that had been collected by Central African were despatched for analysis. The samples outlined dual gold-in-soil anomalies of about 3.5 km north–south strike extent, and as much as 500 m in width in the south-eastern portion of the permit near the village of Fekola. Both anomalies were tested by a single line of RC drill holes, with the easternmost anomaly returning a significant intercept of low-grade gold mineralization within a wide zone of hydrothermal alteration. Papillon subsequently completed further drill testing of the same gold-in-soil anomalies, which resulted in the discovery hole of the Fekola deposit in December 2010.

Work conducted by Papillon since 2010 has consisted of soil sampling, ground geophysical surveys, rotary air-blast (RAB), air core, RC and core drilling, environmental studies to support the ESIS application, geotechnical and hydrological surveys and water sampling, topographic surveys, metallurgical sampling, upgrading of access roads and the accommodation camp, and preliminary mining studies.

In 2012 a scoping-level study was performed, which indicated sufficiently positive economics under the study assumptions that Papillon proceeded with a pre-feasibility study in 2013. The study was prepared using assumptions and allowances in the Australian Securities Exchange (ASX) listing rules and the 2004 Australasian JORC Code. Results were initially summarized in a report prepared by DRA Mineral Projects (Pty) Ltd (DRA, 2013) in June 2013. An update to DRA (2013) was presented by Papillon in a news release to the ASX on 26 June 2013.

The study was also required by the Government of Mali in support of conversion of the exploration permit to an exploitation license and some of the supporting studies in DRA (2013) were prepared in support of the application for the Project environmental permit. The environmental permit was granted to the Project in May 2013 and the exploitation license in January 2014.

The Papillion PFS indicated positive project economics under the assumptions in the study. As a consequence, Papillon commenced more detailed engineering and technical studies, and had undertaken some preliminary site works in support of future project construction, including site-clearing activities.

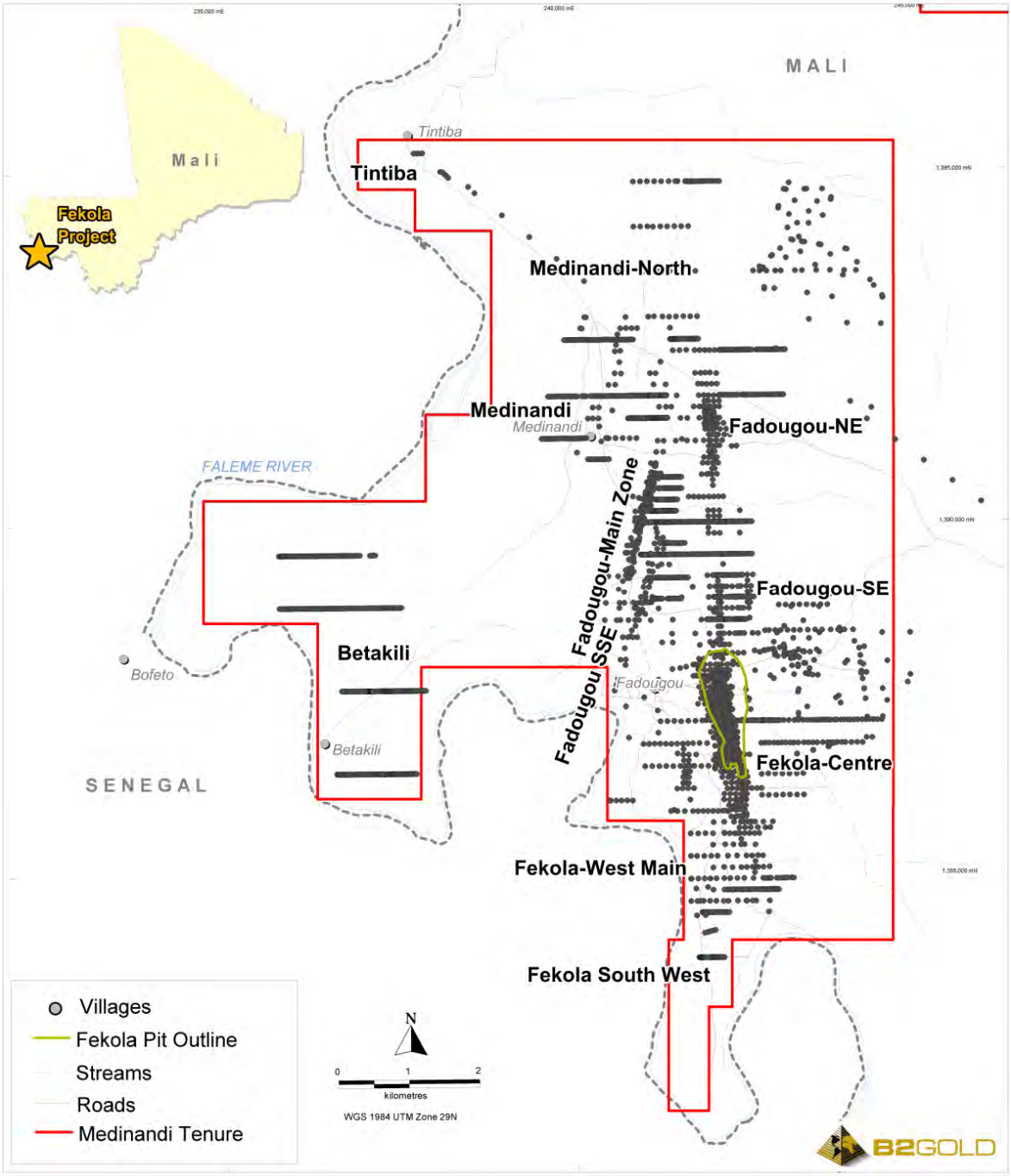
Table 6.2.1 summarizes the available information on the exploration and work programs completed prior to B2Gold's acquiring Papillon's interests. Prospects and targets identified from this work are shown on Figure 6.2.1.

Table 6.2.1 Summary Table of Historical Activities, Médinandi Exploitation Permit

Company	Year	Work Completed
Prospector	1954 – 1955	Documentation of activities not available to B2Gold.
Sonarem	1962 – 1970	Appears to have included geological mapping, reconnaissance trenching and surface sampling, auger and core drilling, geophysical surveys. BRGM reportedly undertook a resource estimate. Work is not broken out in the information available as to which companies performed which activity. The programs identified a 10 km^2 zone which contained five gold anomalies related to a well-defined structural feature. The two largest zones, noted as II and III, were considered to be potentially economic. The Médinandi gold deposit was interpreted to host three mineralized zones, oriented north–south and northeast–southwest, over an area of 1,200 m x 3.3 m.
BRGM	1975 – 1982	
Guefest	1992 – 1996	
WAG	1997 – 1998	Regional mapping, ground IP surveys, trenching, soil and termite geochemical sampling, auger and reverse circulation (RC) drilling, and resource estimates for the Fadougou Main Zone.
Randgold	1998 – 2001	Interpretation of Landsat and aeromagnetic data, geological and regolith mapping, regional geochemical soil and rock, compilation of data from previous work, and updated mineral resource estimate for the Fadougou deposit.
Central African/Songhoi	2006 – 2009	Mapping, soil geochemical surveys, IP and airborne magnetic and EM surveys over the project area, together with RC and core drilling (130 holes) over the Médinandi and Fadougou zones; updated resource estimate at Fadougou.
Colonial/Papillon/Songhoi	2010	110 RC holes (13,081 m) at Fadougou, Fekola and Tintiba; 162.9 line km of geophysical surveys; 300 termite samples from mounds located between Médinandi and Betakili; 114 RC holes (14,211 m) on 14 prospects around Fadougou and Fekola; 122 RAB holes (3,065 m) at Fekola.
	2011	Resistivity survey for underground potable water; pole–dipole ground geophysics, 17 east–west-oriented lines, with stations on 160 x 1,600 m spacing at Fekola; 1,546 soil samples at Médinandi East, Fadougou East, Fekola East; 107 aircore holes (1,782 m) at Betakili; 986 RAB holes (20,100 m) at Fadougou East, Fadougou NE, Fadougou ESE, and Betakili; 181 RC holes (12,449 m) at Fadougou E, Fadougou SE, Fadougou Central, Fekola; 14 core holes (2747.5 m) at Fekola.
	2012	1,132 soil samples at Médinandi north; ground geophysical surveys northwest of Médinandi and at Fekola; test gravimetric survey; collection of metallurgical samples for testwork; road access, camp and electrical supply upgrades; installation of automatic weather station; condemnation and sterilization drilling (59 holes for 6,325 m at Fadougou and Fekola); water bore drilling; LiDAR survey; geotechnical and environmental studies; 81 RC holes (10,271 m) at Fadougou SE and Fekola; 179 core holes (45,452 m); completion of a scoping-level study.

Company	Year	Work Completed
	2013	Ground gravimetric test survey at Fekola north; road access and camp upgrades; continuation of environmental and geotechnical studies (including 20 core holes, 15 RC holes and 48 pits completed for geotechnical purposes); 18 boreholes (1,262 m) of hydrological drilling; one batch of 669 core samples (approximately 964.8 kg) and second batch of 822 core samples (approximately 1219.3 kg) collected and prepared for additional metallurgical testwork; 37 core holes (11,768 m) at Fekola, project-wide drilling, including geotechnical holes totalling 274 holes (31,414 m) at Fekola and Fadougou SE; receipt on site of the run-of-mine laboratory equipment and container; completion of a pre-feasibility study; environmental permit granted. Resource estimate for Fekola deposit supporting the Papillon PFS released to ASX on 24 January 2014. Updated resource estimate for the Fekola deposit released to the ASX on 3 September 2013.
	Q1 2014	Water sampling; 84 exploration drill holes (7,902 m); four geotechnical boreholes and 62 test geotechnical pits completed in the proposed TSF and plant areas, additional geotechnical work in the area of the east wall of the proposed pit, packer testing; XRF analysis of selected core intervals from five core holes; magnetic susceptibility measurements on 18 RC holes (5,940 samples); commencement of studies in support of detailed design; grant of exploitation license. Resource estimate for Fekola Satellites (Fadougou) updated.

Figure 6.2.1 Prospect and Deposit Location Plan, Médinandi Exploitation Permit



6.3 Resource History

Two relevant historical estimates were prepared by Papillon on the Fekola deposit, prior to B2Gold acquiring Papillon. As part of the Papillon PFS, a mineral resource estimate was completed on the Fekola deposit by MPR, and was provided to the ASX as part of Papillon's ongoing disclosure obligations to the ASX on 24 January 2013.

Post the completion of the Papillon PFS, additional drilling was undertaken at Fekola and surrounding exploration targets. This resulted in an updated Mineral Resource estimate for Fekola that was completed by MPR in August 2013, and released by Papillon to the ASX on 3 September 2013.

The historical estimates are summarized in Table 6.3.1 (January 2013) and Table 6.3.2 (August 2013). The notes to the tables which follow Table 6.3.1 and Table 6.3.2 provide the context for the historical estimates. B2Gold is not treating the historical estimates as current mineral resources as the Mineral Resources reported in Section 14 are considered to supersede the Papillon estimates.

The QPs have used the MPR resource models, DRA (2013), the Papillon PFS, internal Papillon studies and, where applicable, interim results of work undertaken since the completion of the studies as supporting documentation when preparing the financial analysis and the Mineral Resource estimates presented in this Report.

Table 6.3.1 January 2013 Papillon Historical Resource Estimate

Lower Cut off Grade (g/t Au)	Measured			Indicated			Inferred		
	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold Moz)
0.2	96.1	1.23	3.79	42.7	0.89	1.22	54.7	0.8	1.3
0.3	83.0	1.38	3.68	33.4	1.07	1.15	40.0	1.0	1.2
0.4	72.5	1.53	3.57	26.9	1.24	1.08	31.5	1.1	1.1
0.5	63.5	1.68	3.44	21.7	1.43	1.00	25.4	1.3	1.0
0.6	55.7	1.84	3.30	17.9	1.62	0.93	20.6	1.4	1.0
0.7	49.1	2.00	3.16	15.0	1.81	0.87	17.0	1.6	0.9
0.8	43.5	2.16	3.03	12.7	2.00	0.82	14.3	1.8	0.8
0.9	38.8	2.32	2.90	11.0	2.19	0.77	12.2	1.9	0.8
1.0	34.7	2.48	2.77	9.6	2.36	0.73	10.7	2.1	0.7
1.1	31.3	2.64	2.66	8.5	2.54	0.69	9.4	2.2	0.7
1.2	28.2	2.80	2.54	7.6	2.70	0.66	8.3	2.4	0.6
1.5	21.4	3.27	2.25	5.6	3.18	0.57	6.0	2.8	0.5

Base case is highlighted

Table 6.3.2 August 2013 Papillon Historical Resource Estimate

Lower Cut off Grade (g/t Au)	Measured			Indicated			Inferred		
	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (Moz)
0.2	120.09	1.15	4.44	77.40	0.93	2.31	69.0	0.59	1.3
0.3	102.53	1.31	4.32	59.87	1.14	2.19	45.3	0.77	1.1
0.4	88.38	1.46	4.15	48.32	1.32	2.05	32.1	0.95	1.0
0.5	76.31	1.62	3.97	39.98	1.51	1.94	24.1	1.11	0.9
0.6	66.33	1.78	3.80	33.68	1.69	1.83	18.5	1.28	0.8
0.7	57.98	1.95	3.64	28.87	1.86	1.73	14.7	1.45	0.7
0.8	51.10	2.11	3.47	25.08	2.03	1.64	11.9	1.61	0.6
0.9	45.32	2.27	3.31	22.05	2.19	1.55	9.9	1.77	0.6
1.0	40.44	2.43	3.16	19.57	2.35	1.48	8.3	1.33	0.5
1.1	36.29	2.59	3.02	17.51	2.5	1.41	7.0	2.08	0.5
1.2	32.72	2.74	2.88	15.79	2.65	1.34	6.0	2.23	0.4
1.5	24.64	3.2	2.53	11.99	3.06	1.18	4.0	2.68	0.3

Notes to accompany historical estimate tables:

1. **Base case is highlighted**
2. **Mineral Resource estimates were prepared by Mr Nic Johnson, M.AIG, an employee of MPR Geological Consultants Pty Ltd and are reported on a 100% basis. The January 2013 estimate has an effective date of 15 January, 2013 and was released to the ASX on 24 January 2013; the September 2013 estimate has an effective date of 5 August 2013 and was released to the ASX on 3 September 2013.**
3. **Resources are reported using a base case cutoff grade of 1 g/t Au.**
4. **Estimates were performed using core and RC drill data. The January estimate is supported by 428 RC, RC pre-collar and diamond core (RC-DD), and diamond core drill holes for a combined length of 80,968.9 m; 79,723 assay sample intervals predominantly sampled at 1 m intervals downhole; and 1,125 bulk density measurements. The August estimate is supported by 643 RC, RC-DD and core holes for a combined length of 127,148.40 m; 125,365 assay sample intervals predominantly sampled at 1 m intervals downhole; and 2,003 bulk density measurements.**
5. **Geological and oxidation interpretations were imposed to define domains of similar grade tenor and directional trends. The models estimate resources into panels with dimensions of 20 mE x 40 mN x 5 mRL. Multiple Indicator Kriging (MIK) of gold grades used indicator variography based on 2 m composite grades, with continuity of gold grades characterised by variograms at 14 indicator thresholds. A panel support adjustment, incorporating an adjustment for information effects, was used in estimation, assuming a selective mining unit of 5 mE x 10 mN x 5 mRL and a nominal grade control sampling spacing of 8 mE x 15 mN x 1.5 mRL. The shape of the local panel gold grade distribution was assumed to be lognormal within each panel as estimated by indicator kriging.**

6. **The "recoverable resource" estimates within each panel were classified according to the distribution of sampling in the kriging neighbourhood. The classification scheme was related to the proximity and spatial distribution of the informing sample composites. Measured and Indicated Mineral Resources assume at least a 40 x 50 m spacing; Inferred Mineral Resources have drill spacings of at least 80 x 160 m.**
7. **The resources are reported using the classifications of the Australasian JORC Code, 2004 edition. While these codes are generally reconcilable to the CIM Definition Standards, B2Gold is not treating the estimates as meeting CIM standards as they have not been evaluated for reasonable prospects of economic extraction, such as constraining within a Lerchs–Grossmann pit shell; instead they are reported only within a marginal cutoff grade.**
8. **B2Gold considers these estimates to be relevant to the Project history as the January 2013 estimate is the basis for the Papillon PFS, and the August 2013 estimate reflects Papillon's latest public disclosure to the ASX on the Fekola deposit prior to the B2Gold/Papillon transaction.**
9. **B2Gold is not treating the historical estimates as current. A QP has not done sufficient work to classify the historical estimates as current Mineral Resources. B2Gold is reporting updated resource estimates in Section 14 of the Report that are considered to supersede the Papillon estimates.**

FEKOLA GOLD PROJECT

NI 43-101 - GEOLOGICAL SETTING, MINERALIZATION AND ALTERATION

3174-STY-002

Table of Contents

			Page
7.0	**GEOLOGICAL SETTING, MINERALIZATION AND ALTERATION**		**7.1**
7.1	Regional Geology		7.1
	7.1.1	Overview	7.1
	7.1.2	Regional Stratigraphy	7.1
	7.1.3	Regional Structures	7.3
7.2	Deposit Geology		7.3
	7.2.2	Deposit Lithology	7.5
7.3	Deposit Scale Geology		7.8
	7.3.1	D_1 and D_2 Structures	7.8
	7.3.2	Brittle-ductile Structures	7.11
	7.3.3	Late Brittle Structures – Fekola Fault	7.13
7.4	Mineralization and Alteration		7.14

FIGURES

Figure 7.1.1	Geology of the Kédougou–Kéniéba Inlier (modified from Lawrence, et al, 2013)	7.2
Figure 7.2.1	Plan map of Fekola Deposit Geology.	7.4
Figure 7.2.2	Principal Lithologies of the Fekola Deposit.	7.6
Figure 7.3.1	Laminated Siltstone / Mudstone Highly Transposed by S_1 Foliation into Strong S_0 / S_1 Composite Fabric	7.8
Figure 7.3.2	Isoclinal F_1 Intrafolial Folds in Layers of Siltstone and Mafic Volcaniclastic(?), FKD_154, 223.1 m	7.9
Figure 7.3.3	F_2 Folds in Banded Siltstone and Mudstone of the Turbidite Unit	7.9
Figure 7.3.4	Equal Area Plot of Poles to Bedding (BED), Contacts (CNT), Fold Surfaces (FOD) and Foliation (FOL)	7.10
Figure 7.3.5	North-looking Vertical Cross Section through the Fekola Deposit, Section 1387397 N	7.11
Figure 7.3.6	Hydrothermally Altered Cataclastic Breccia FKRD_11, 208m	7.12
Figure 7.3.7	Breccia Textured Dolomite-Albite-Quartz Veining, FKD_135, 270.8 and 271.3m	7.12
Figure 7.3.8	Late Brittle Structures Associated with the Fekola Fault Zone	7.13
Figure 7.4.1	Veinlet (A) and Disseminated-style (B) Pyrite Mineralization in Fine Grained Siltstone / Mudstone of the Turbidite Unit	7.15
Figure 7.4.2	East-west Profile of ICP-MS Analyses from ½ HQ Diameter Drill Core (n=1110)	7.15

7.0 GEOLOGICAL SETTING, MINERALIZATION AND ALTERATION

7.1 Regional Geology

7.1.1 Overview

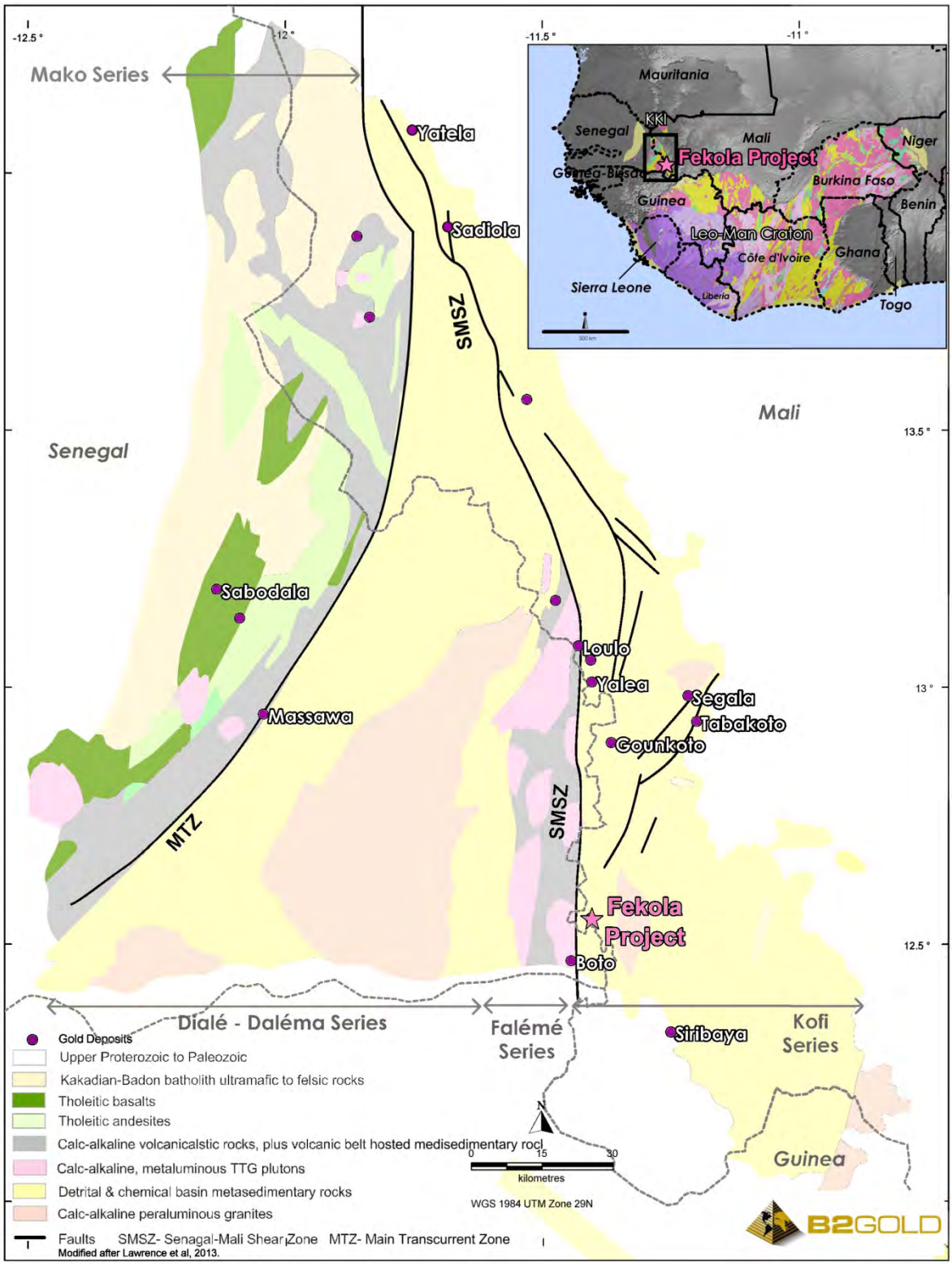
The Fekola deposit is located in the eastern portion of the Paleoproterozoic Kédougou–Kéniéba Inlier ("KKI"), an erosional window through the Neoproterozoic sedimentary cover sequence of the Taoudeni Basin into the Lower Proterozoic volcano-sedimentary and igneous basement rocks of the Birimian Supergroup of the West African Craton (Figure 7.1.1). The inlier is bounded on its western margin by the Hercynian Mauritanide orogenic belt and is unconformably overlain by flat-lying Neoproterozoic sandstones of the Taoudeni intracratonic basin to the east, north and south (Villeneuve and Cornée, 1994).

7.1.2 Regional Stratigraphy

Bassot (1987) proposed a subdivision of the KKI into the Mako Series (granite-greenstone) and the eastern Dialé–Daléma Series (sedimentary basin), separated by the arcuate, north easterly trending Main Transcurrent Zone. The Mako Series comprises tholeiitic basalt and andesite lavas, with intercalated volcanic agglomerates and banded tuffs (Gueye et al., 2008). Calc-alkaline volcaniclastic units and volcanic-rich sedimentary rocks crop out along the eastern side of the Mako Series. The Mako Series has been intruded by a complex sequence of pre- and syntectonic mafic and felsic plutons (Dia et al., 1997). To the east of the Main Transcurrent Zone, the Dialé–Daléma Series consists of sandstones, and siltstones with intercalated calc-alkaline ashfall and lapilli tuffs. It is intruded by a quartz monzonite granite body, the Saraya Granite.

In the eastern Dialé–Daléma basin, a calc-alkaline volcanic-plutonic complex, the Falémé Series may constitute a separate, younger assemblage from the Dialé–Daléma sedimentary basin. The Falémé Series consists of carbonate-rich sedimentary rocks, minor basalts and andesites, rare rhyolites, and syntectonic granitoids (Hirdes and Davis, 2002). The Falémé Series is bounded to the east by the Senegal-Mali Shear Zone, separating it from the Kofi Series sedimentary basin. The Kofi basin comprises sandstones, argillites and platform carbonates and syntectonic, S-type, peraluminous biotite-bearing granites (with similar compositions to the Saraya batholith) (Lawrence et al, 2013). The Kofi Series hosts significant gold mineralization on the eastern side of the Senegal-Mali Shear Zone and is the primary host to mineralization at Fekola.

Figure 7.1.1 Geology of the Kédougou–Kéniéba Inlier (modified from Lawrence, et al, 2013)



7.1.3 Regional Structures

Within the KKI, two major terrane-bounding structures are recognized: the Main Transcurrent Zone and the Senegal-Mali Shear Zone. The Main Transcurrent Zone is an arcuate, north to north easterly trending, crustal-scale shear zone that separates the Mako Series granite–greenstone belt from the younger Dialé–Daléma Series (sedimentary basin) to the east. The Main Transcurrent Zone is spatially related to significant gold mineralization in the Mako Series of Senegal. The northernmost portion of the Main Transcurrent Zone converges and may merge with the north-northwest trending Senegal-Mali Shear Zone.

Senegal-Mali Shear Zone is a crustal scale, terrane bounding shear zone with over 100 km of known strike extent. The Senegal-Mali Shear Zone is spatially associated with significant gold mineralization in western Mali, including the Loulo group of deposits, approximately 50 km north of Fekola, as well as Sadiola and Yatela, located on its northernmost extents.

7.2 Deposit Geology

The Fekola deposit is hosted by a moderate to steeply west dipping, folded sequence of marine meta-sediments of the Kofi group, which locally include: argillite and minor pelitic sediments; fine grained turbidites, comprising laminated to thin-bedded siliciclastic siltstone and mudstone; and a heterolithic, mass flow breccia, or conglomerate (Figure 7.2.1). Fine grained thin marble units are present as a volumetrically minor rock type interbedded with the politic units. Minor mafic volcanic, or volcaniclastic units occur locally in the upper and possibly, lower portions of the hanging wall stratigraphy. Weakly feldspar-phyric felsic dykes are locally observed. The deposit has been subjected to upper greenschist facies metamorphism.

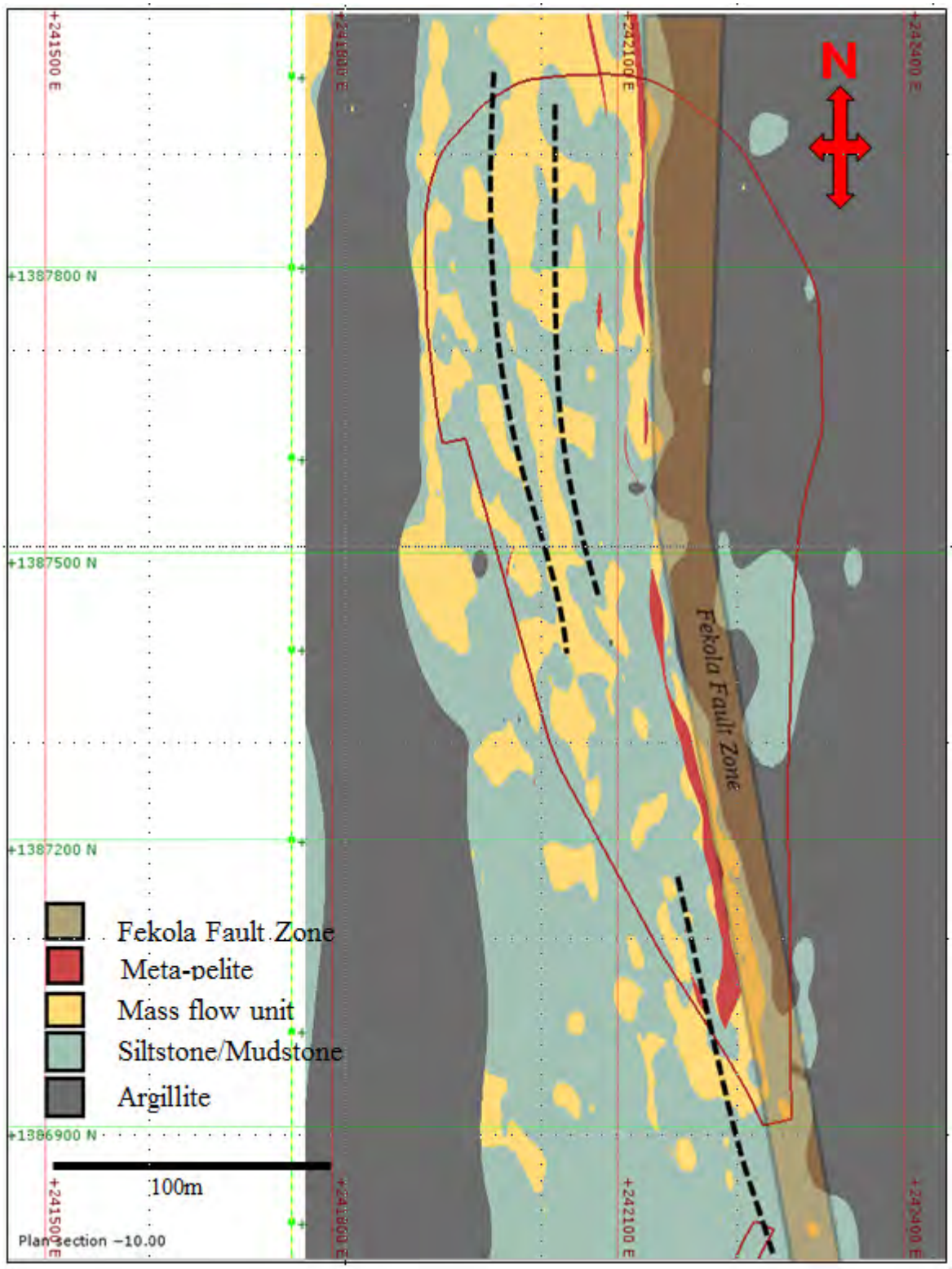
Gold mineralization is associated with fine grained, disseminated pyrite and local pyrite veinlets. Mineralization is also associated with a pervasive, texturally destructive, hydrothermal dolomite alteration, which may overprint an earlier albite-quartz-carbonate±sericite alteration. Hydrothermal alteration and gold mineralization appear to be synchronous with the development of north to north-northwest plunging folds within a corridor of ductile, high strain deformation.

The late-stage, brittle fault zone known as the Fekola Fault, cross cuts all earlier generations of structures and mineralization. The magnitude and sense of displacement along this late structure are currently unknown.

The Fekola deposit area is covered by a laterally extensive, in-situ and transported lateritic regolith that measures between 15 and 45 m. The regolith profile is overlain by a thin cover of soil. In areas of transported cover, the top of the weathering profile typically consists of a mottled clay zone, comprising ferruginous, and grey clays. This grades downward into an alluvial polymictic cobble / gravel unit that is set in a clay matrix. Underlying the alluvial unit is a residual laterite profile that is typical of the downward succession seen in tropical and weathered terrains, which, from surface, transitions from saprolite to saprock to fresh rock. The mineralisation at the Fekola deposit is extensively covered by the layer of alluvium, with the exception of the southernmost portion of the deposit, where residual laterite occurs at surface.

The majority of gold mineralization in the Fekola deposit occurs in unweathered, fresh rock.

Figure 7.2.1 Plan map of Fekola Deposit Geology.



Deposit area is underlain by west dipping pelitic rocks. Late-stage Fekola Fault Zone (shown in brown) is parallel in strike to mineralization. Proposed pit outline shown in red. Dashed lines indicate inferred trace of D2 structures.

7.2.2 Deposit Lithology

As part of a recent re-modelling of geology by B2Gold Corp, a simplified stratigraphy has been devised, in order to better facilitate three-dimensional rendering of the lithologic units. Examples of the main lithologies of the revised model are shown in Figure 7.2.2. A geological drill core re-logging program is planned for 2015, in order to provide greater resolution of the deposit stratigraphy.

Argillite

Argillite is project terminology describing a chloritic phyllite (Figure 7.2.2 A and D) that comprises the hanging wall and footwall lithologies to the mineralized turbidite / debris flow package at Fekola. Footwall argillite, particularly in the immediate footwall of the mineralized zone, tends to exhibit bands of dark green chlorite alteration (Figure 7.2.2 D). The argillite package may also include intervals of massive, dark green mafic volcanic rock. Total thickness of the argillite unit is unknown, but it is locally intersected in drill hole to widths of greater than 50 m.

Turbidite (Banded Siltstone and Mudstone)

The most abundant rock type by volume in the Fekola deposit, the main turbidite unit comprises thinly bedded and locally, laminated siltstones and mudstones (Figure 7.2.2 B), with local horizons of fine grained, light grey sandstone, or greywacke. Colouration ranges from pale yellow-tan to orangey-pink with local greyish intervals. Petrography indicates that the turbidites can have a high carbonate content, possibly secondary dolomite ranging up to >80% by volume, which locally obscures the protolith. Individual beds and laminae exhibit isoclinal folding on the meso- and macroscopic scale and primary bedding is frequently highly transposed by overprinting fabrics. Boyd, et al. (2013) report local occurrence of graded bedding in the turbidite package. Younging directions are difficult to discern, due to the high degree of transposition by later deformation. Minor mafic intrusions also occur within the turbidite unit. Turbidites have been intersected in drillhole over widths of >100 m. The true thickness of this package is currently unknown.

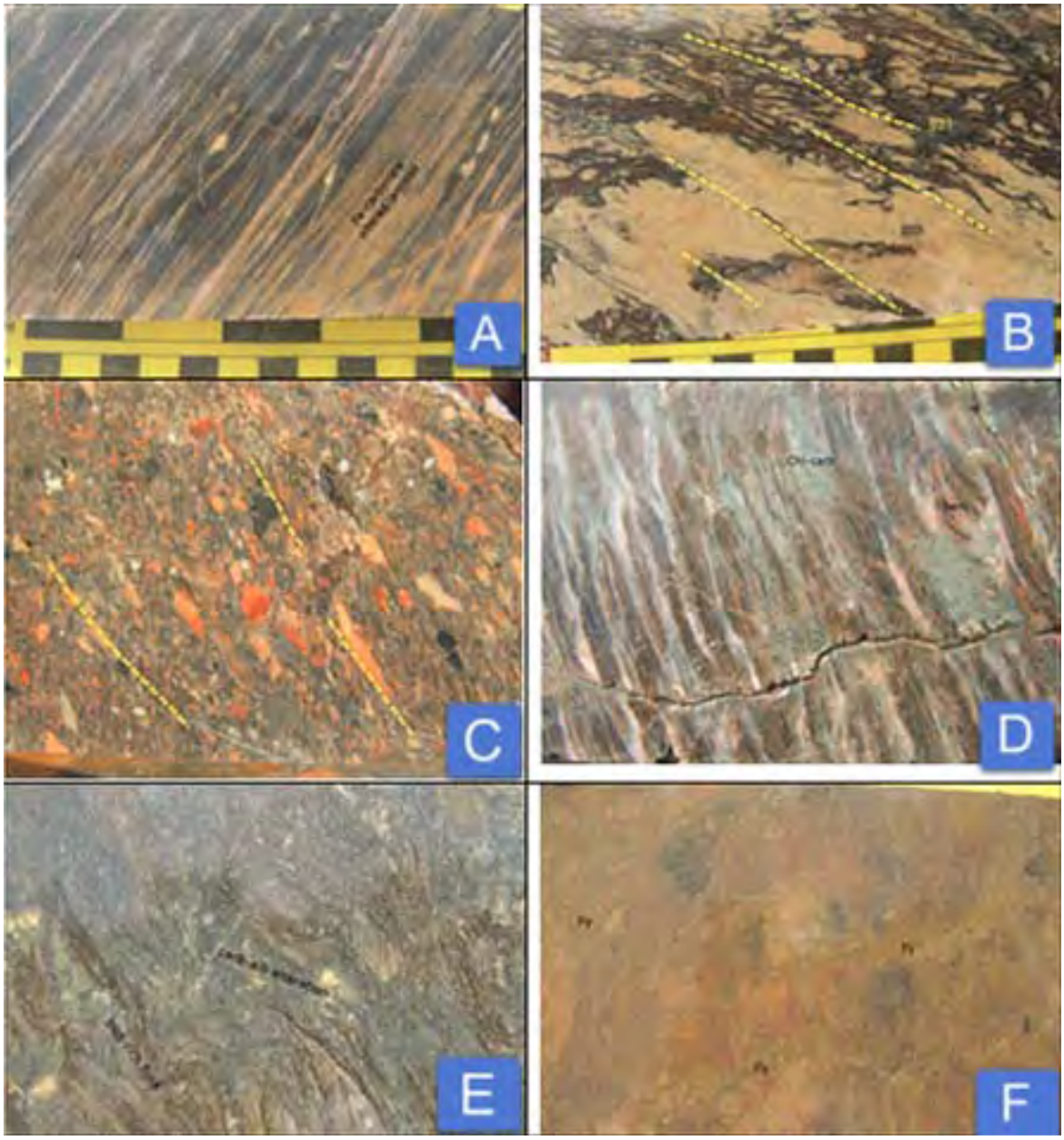
Mass Flow Breccia

The term mass flow breccia, mass debris flow and meta-conglomerate have been variably used at Fekola to describe a heterolithic, dominantly matrix supported sedimentary breccia and conglomerate comprised of subangular, pebble-sized fragments, or clasts. The fragments themselves are locally highly deformed. Typically, the mass flow has a diverse mineral assemblage reflecting the variety of fragments. Colouration is varied and reflects the abundance of clasts of varying composition.

Detailed analysis reported by Boyd et al (2013) documents the presence of fining-upward cycles ranging in thickness from less than a meter to several meters. Coarser fragmental intervals are generally <25% of the entire interval and are locally capped by an interval of grey, very fine to fine-grained quartz arenites and subordinate greywackes, with thin beds of graded and laminated silty mudstones. Younging directions determined from graded bedding are frequently difficult to discern due to overprinting by subsequent deformation.

Petrographic analysis (Ross, 2014) shows that both fragments and the surrounding matrix have a foliation fabric. Rare fragments may be quartz porphyritic and the matrix locally contains prismatic quartz crystals which may be liberated phenocrysts. Isolated, rhombic carbonate crystals are also locally observed. Fragments are variably albite, quartz, muscovite, biotite or carbonate altered. Trace amounts of tourmaline and possibly chlorite are also present in the matrix. Carbonate is the dominant alteration mineral (dolomite) and can strongly overprint clasts and forming a cloudy brownish matrix, possibly mixed with biotite. In areas of strongest dolomitization, primary textures are almost completely obliterated (Figure 7.2.2 F). True thickness of the mass flow breccia unit is unknown, though diamond drill intersections of this unit can locally exceed 50 m.

Figure 7.2.2 Principal Lithologies of the Fekola Deposit.



A) hematite stained argillite (hanging wall); B) laminated to thin-bedded siltstones and mudstones of the Turbidite package; C) heterolithic pebble conglomerate, or mass flow breccia; D) strongly chlorite altered argillite (footwall); E) metapelite and F) example of mineralized mass flow breccia. Dashed yellow lines indicate foliation/cleavage direction in hand specimen. All samples from ½ HQ diameter drill core. Centimetre scale indicated in A and B.

Metapelite

Metapelite is the current project term describing a strongly foliated, relatively biotite-rich rock (Figure 7.2.2 E). Historically, this rock has been variably described as metamorphosed arkose, or alternatively, as a mafic intrusive. In hand specimen, the metapelite is dark grey-black with wispy pale folia (Figure 7.2.2 E). It is locally, moderately magnetic, but magnetite is not readily apparent in hand specimen. Sulphide content can range from minor amounts to >5% by volume. When mineralized it typically exhibits very fine-grained irregularly distributed, disseminated pyrite. It can be an excellent host rock to gold mineralization. The metapelite is most frequently observed within several tens of metres of the hanging wall of the Fekola Fault, in drillhole intersections ranging from less than a metre up to approximately 25 m thick. It locally exhibits a lenticular geometry in plan and cross sectional views, which may reflect local structural thickening or conversely, attenuation of this unit.

Petrographic analysis (Ross, 2014) indicates that the metapelite is comprised of anastomosing biotite folia wrapping around domains of albite-carbonate altered rock.

The protolith to this rock may have been metasedimentary, or igneous, comprising alternating biotite-feldspar laminae. The feldspar component has been overprinted by albite-carbonate alteration. Carbonate alteration may be originally synchronous with albite, but outlasts it and partially overprints it. The biotite is relatively unaltered. The biotite folia are crenulated, suggesting that the metapelite has undergone at least two stages of ductile deformation. Albite-carbonate alteration appears to post-date ductile deformation.

Marble

The marble is dominantly white to pale grey and finely laminated and is comprised either of calcite, or of interlocking semi-rhombic dolomitic carbonate (88%). The laminations are tectonic and defined by variations in carbonate grain size (<20 um – 0.2 mm) and a few very fine-grained quartz dominant folia which wrap around porphyroclasts defining asymmetric mylonitic fabrics. Randomly oriented flakes of muscovite (2%) occur in the carbonate. Locally, there are trace amounts of nearly microcrystalline pyrite disseminated in the rock. The predominant carbonate is likely dolomite (Ross, 2014) close to mineralized areas, but calcite more distally.

This rock is variably mineralized, but not a significant host to gold mineralization at Fekola. In drill core, this unit occurs over intervals of several decimetres, to thicknesses in excess of 20 m. The marble occurs intermittently throughout the stratigraphy, likely as a result of repetition of individual units by F_2 folds. As with the metapelite unit, it is suspected to have undergone local structural thickening and or attenuation during deformation. For the purposes of three-dimensional modelling, and since it often occurs in a similar stratigraphic position, the marble has been grouped into the larger mass flow breccia unit. Boyd et al (2013) interpret the calcareous unit as capping the coarse, clastic debris flows.

Felsic dykes or sills

Pink, weakly feldspar ± quartz porphyritic dykes or sills which have a pink, aphanitic to albitized matrix, and which locally exceed 10 m in thickness occur within the turbidite and mass flow units. This unit has historically been recorded as a hematite altered sandstone, but the sharp contacts

and porphyritic texture are inconsistent with this classification. Further resolution of the distribution and geometry of these dykes is expected via the 2015 re-logging program.

7.3 Deposit Scale Geology

A thorough interpretation and discussion of the structural geology of the Fekola deposit is presented in technical paper by Dorling (2013). B2Gold Corp is currently revising and re-interpreting this earlier work, with a detailed structural analysis to be completed in 2015. The Fekola deposit is hosted within a north to north-northwest trending zone of intense ductile deformation.

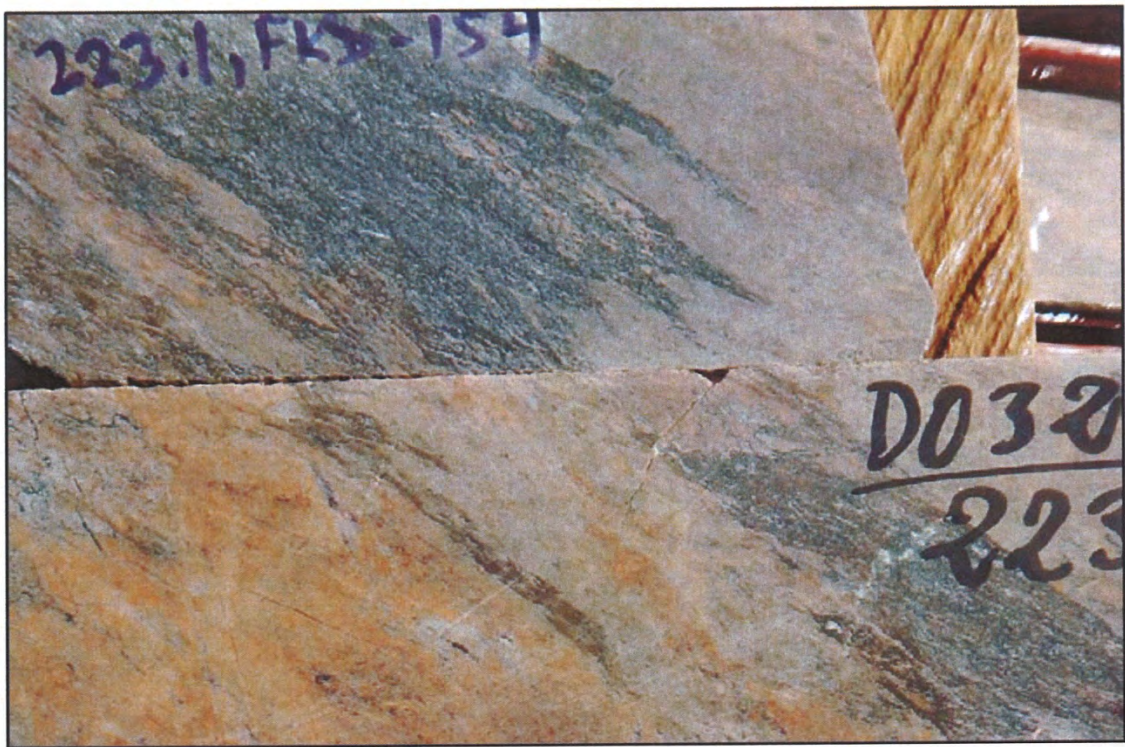
7.3.1 D_1 and D_2 Structures

D_1 is characterized by a well-developed penetrative fabric termed S_1. S_1 is an intense foliation, defined by alignment of phyllosilicate minerals, including sericite, chlorite and locally, biotite. In intensely deformed turbiditic rocks, S_1 is defined by lenticular domains of strongly transposed S_0 laminae that are wrapped by spaced phyllosilicate folia. S_1 frequently transposes bedding into a strong, S_0 / S_1 composite fabric (Figure 7.3.1). S_1 is locally axial planar to isoclinal, steeply inclined, intrafolial F_1 folds (Figure 7.3.2).

Figure 7.3.1 Laminated Siltstone / Mudstone Highly Transposed by S_1 Foliation into Strong S_0 / S_1 Composite Fabric

Examples from FKD_139, 228.7m and 229.6m.

Figure 7.3.2 Isoclinal F_1 Intrafolial Folds in Layers of Siltstone and Mafic Volcaniclastic(?), FKD_154, 223.1 m



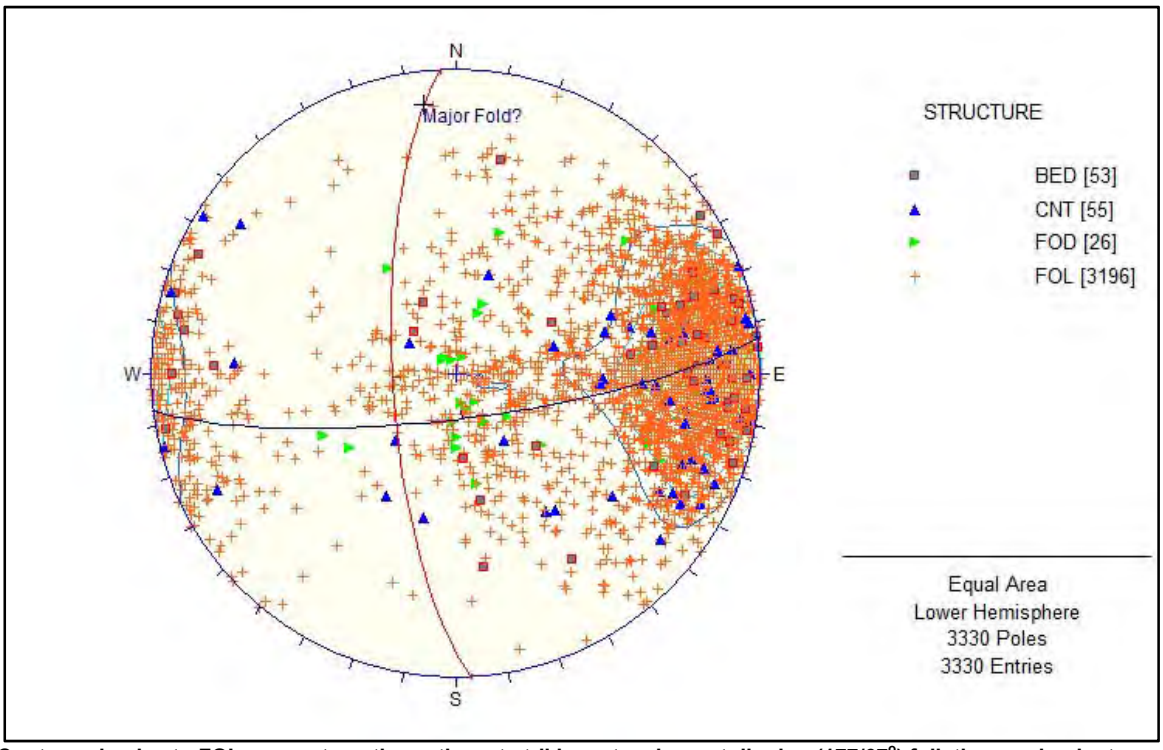
D_2 structures are characterized by open to close, F_2 folds (Figure 7.3.3), which overprint S_1 and are coaxial to F_1 folds. Locally, tight F_2 kink folds are observed, particularly within thin-bedded to laminated sediments of the turbidite unit.

Figure 7.3.3 F_2 Folds in Banded Siltstone and Mudstone of the Turbidite Unit

A) open to B) tight folding, FKRD_033, 138.6-140m; C) and D) tight folding, FKD_139, 338.7m, and E) tight, kink-style folds, FKD_150, 231.3m. Trace of S_0/S_1 fabric shown in black dash (A and B).

An analysis of planar fabrics measured in oriented drill core by Papillion Resources, shows the predominant southerly strike and steep westerly dips that characterize S_1 at Fekola. Figure 7.3.4 shows an equal area stereonet plot of poles to measured planar elements. The combined contour peak for bedding, contacts and foliation indicates a mean strike and dip of 177/67. Significant scatter of the poles, particularly to foliation, in the centre of the net indicate relatively flat lying fabrics and may reflect reorientation of S_1 by F_2 folding. The pole to a plane fitted through this scattered foliation and bedding data approximates the shallowly north-northwest plunging (13°→354) axis of the major F_2 folding event that has produced the current orientation of fabrics.

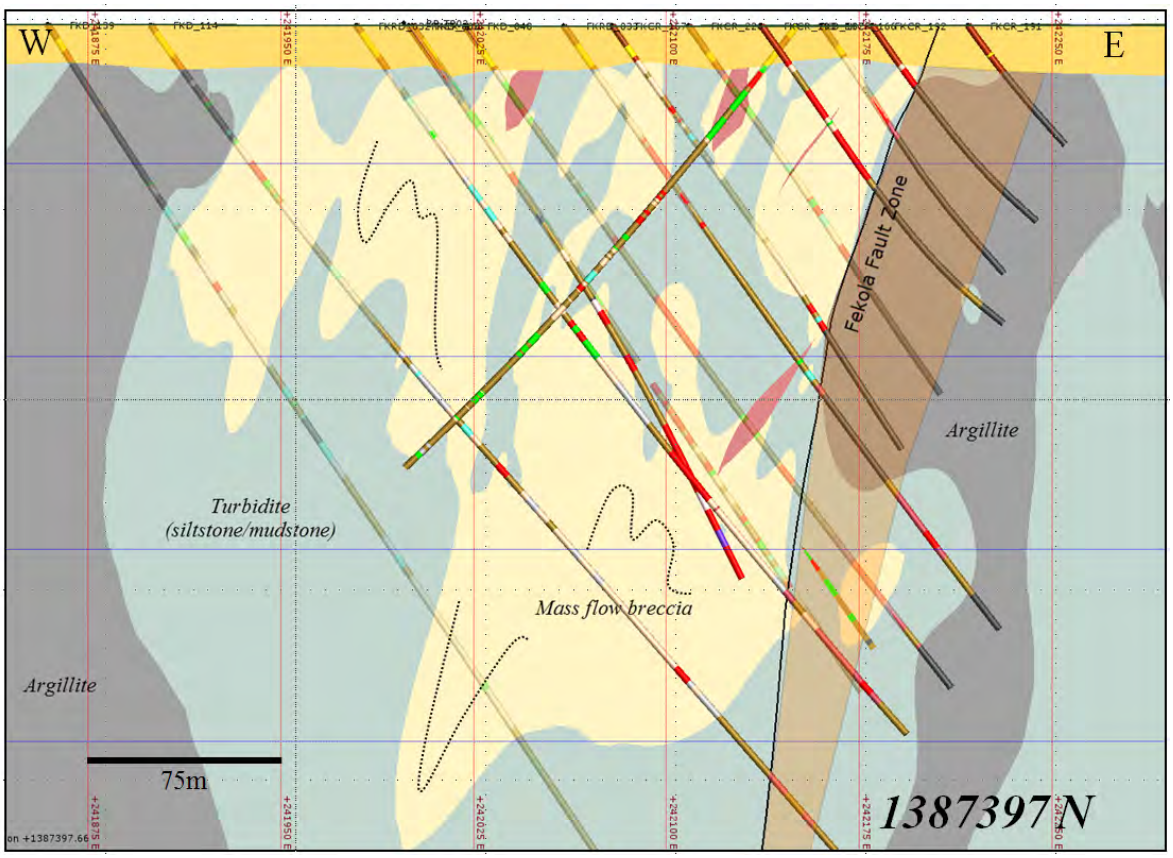
Figure 7.3.4 Equal Area Plot of Poles to Bedding (BED), Contacts (CNT), Fold Surfaces (FOD) and Foliation (FOL)



Contoured poles to FOL suggest south-southwest striking, steeply west dipping (177/67°) foliation predominates, although there is a loose cluster of poles at the centre of the net that may indicate an earlier S_0/S_2 composite foliation has been reoriented by F_2 folding. The subtle, easterly trending girdle of poles to bedding surfaces (blue great circle), including CNT and FOD, suggests shallowly north-northwest plunging F_2 folding (13°→354).

On vertical cross-sections through the deposit, folding is characterized by asymmetric, apparently east-verging, antiform-synform pairs. These smaller scale macroscopic folds may occupy a larger synformal structure in the area of the central, thickest portions of the Fekola deposit, which is inferred from the geometry of the enveloping surfaces of folded stratigraphic units and the repetition of argillite units in the footwall and hanging wall of the deposit (Figure 7.3.5). Additional structural work is required to confirm this interpretation.

Figure 7.3.5 North-looking Vertical Cross Section through the Fekola Deposit, Section 1387397 N



At the deposit scale, stratigraphy exhibits asymmetric, east-verging folds with local structural thickening of individual units and corresponding attenuation along fold limbs. The structural late, brittle Fekola Fault zone (brown) crosscuts all stratigraphy and earlier structure. Metapelite unit shown in pale purple fill.

Individual stratigraphic units appear to exhibit structural thickening in the hinge zones of decimetre-scale F_2 folds, as well as significant attenuation along east and west dipping limb regions.

7.3.2 Brittle-ductile Structures

Although deformation at Fekola is predominantly ductile in nature, examples of more brittle-ductile structures are evident in the form of:

- Cataclastic breccias with hydrothermally altered, pyritic matrices (Figure 7.3.6).
- Zones of pale grey, breccia textured dolomite-albite-quartz veining (Figure 7.3.7).

These brittle-ductile structures often are present in high grade domains and are interpreted as being concurrent with ductile deformation at Fekola and may reflect episodic fluid overpressuring in the hydrothermal system.

Figure 7.3.6 Hydrothermally Altered Cataclastic Breccia FKRD_11, 208m

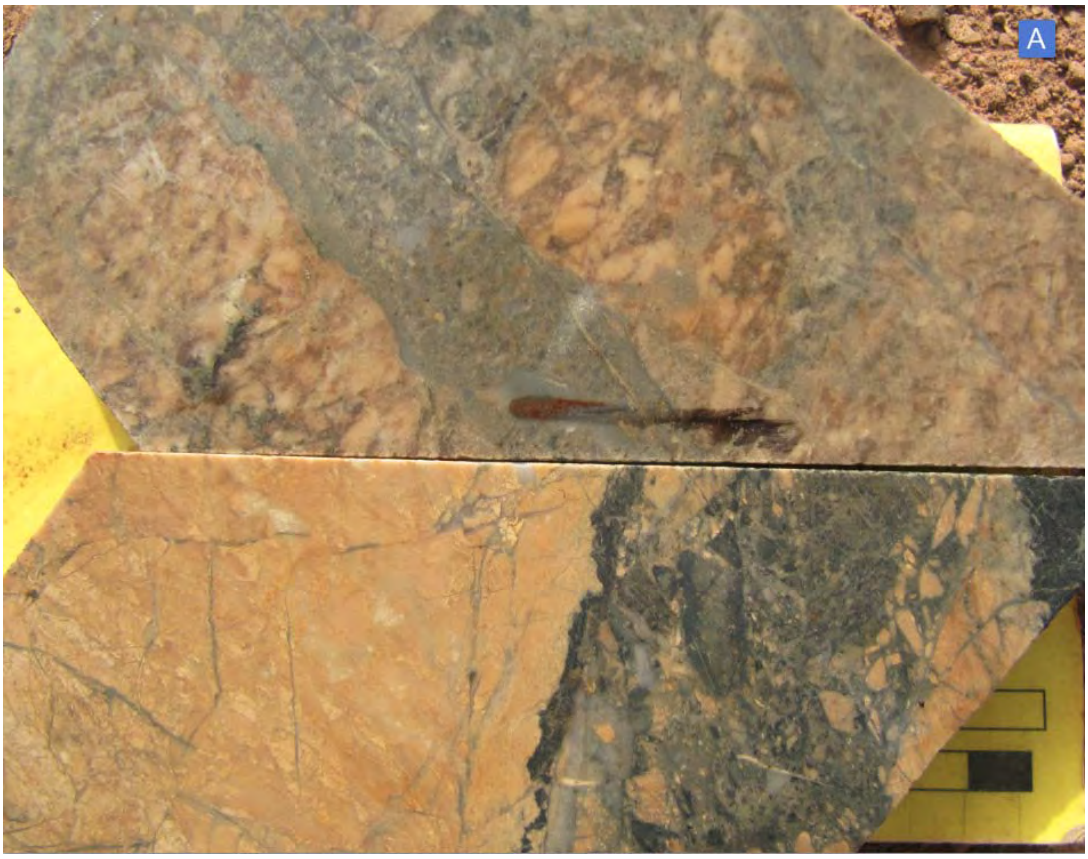
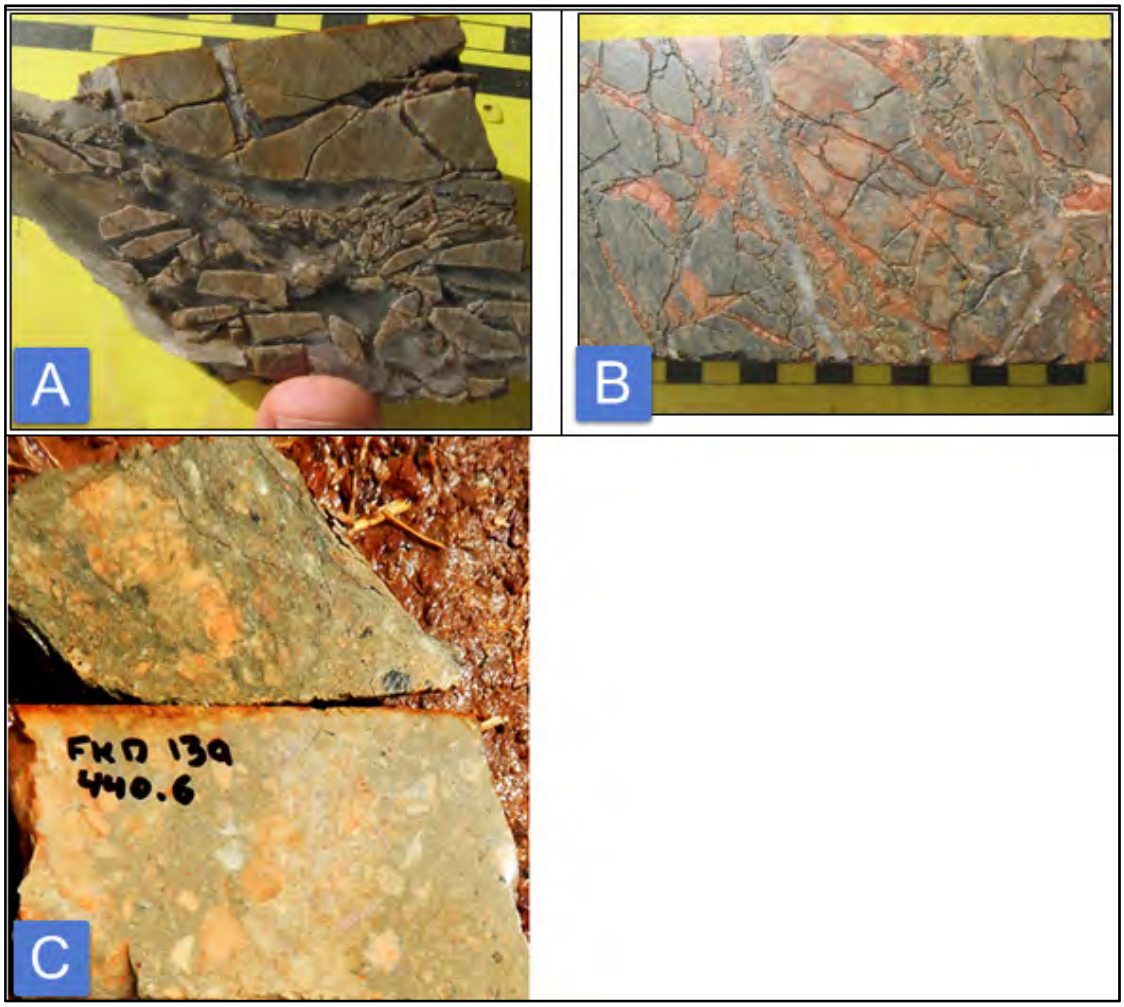


Figure 7.3.7 Breccia Textured Dolomite-Albite-Quartz Veining, FKD_135, 270.8 and 271.3m

7.3.3 Late Brittle Structures – Fekola Fault

The south to south-southeast striking Fekola Fault is the latest significant structure observed in the Fekola deposit. The fault exhibits steep (70 to 80°) westerly dips in the north end of the deposit, with more moderate dips (45 to 50°) prevailing along strike to the south. The fault is characterized by a zone of brittle deformation comprising both lithified and unconsolidated fault breccias, locally chloritic cataclasites, chloritic fault gouge and zones of highly fractured and broken core. Within the zone, fault strands occur tens of meters apart, but in general fault slip zones are <2 m wide, with damage zones having chlorite fractures over a few meters. Fault breccias locally exhibit pristine, mosaic-type textures, cemented by clear gypsum and/or hematite-stained gypsum and calcite matrices (Figure 7.3.8). The direction and magnitude of displacement along the Fekola Fault is currently unknown. It strikes parallel to, and occurs in the lower eastern parts of the deposit.

Figure 7.3.8 Late Brittle Structures Associated with the Fekola Fault Zone



Fault breccias are locally cemented with clear gypsum (A), or pinkish, hematite-stained gypsum-calcite matrices (B). Chloritic cataclastic breccia typical in Fekola Fault (C)

7.4 Mineralization and Alteration

Gold mineralization in the Fekola deposit is associated with fine grained pyrite in tan dolomite-albite altered wallrocks, which locally contain diffuse, often deformed pale grey quartz-dolomite-pyrite-albite matrix veins and veinlets (Figure 7.3.7B). Pyrite occurs both as fine disseminations and within millimetre-scale pyrite veinlets, which have locally been deformed by D_1 (Figure 7.4.1). The folding of zones of pyrite mineralization by F_1 and F_2 folds indicates that that the main mineralization event at Fekola was early in the deformation history of the deposit. This early mineralization event is reflected in the current geometry of the ore zone, which plunges shallowly to the north-north west, in an orientation similar to that of folds and lineations. This plunging zone of mineralization is broadly continuous and has been traced over a strike extent of approximately 1.5 km, to depths of up to 400 m below surface, with widths to 300 m. The greatest continuity is observed within a high grade shoot (>2 g/t Au) which plunges approximately 13° to the north-northwest. Mineralization is open at depth, down plunge. Trace amounts of chalcopyrite have also been observed. The total sulphide content of the deposit is typically less than 5%.

Gold mineralization is proximally associated with intense, texturally destructive, carbonate (dolomite), albite, quartz, and sericite alteration. Dull-grey to tan coloured dolomite is the most abundant alteration mineral at Fekola. Dolomite occurs in the main phase of mineralization within veinlets and the matrix of breccias, but also appears to be part of a longer-lived, pervasive carbonate alteration that locally overprints foliation, earlier alteration and mineralization (Ross, 2015). The extensive carbonate alteration overprints an earlier, pervasive, orangey-pink albite alteration. An early stage of muscovite is associated with the development of the foliation fabrics, but it is mostly overprinted, first by albite-carbonate, then carbonate-pyrite alteration. There is evidence of a minor late stage biotite alteration event, occurring with quartz-carbonate veinlets. Magnetite occurs intermittently throughout the host stratigraphy, but the spatial relationship to gold mineralization is not yet fully understood.

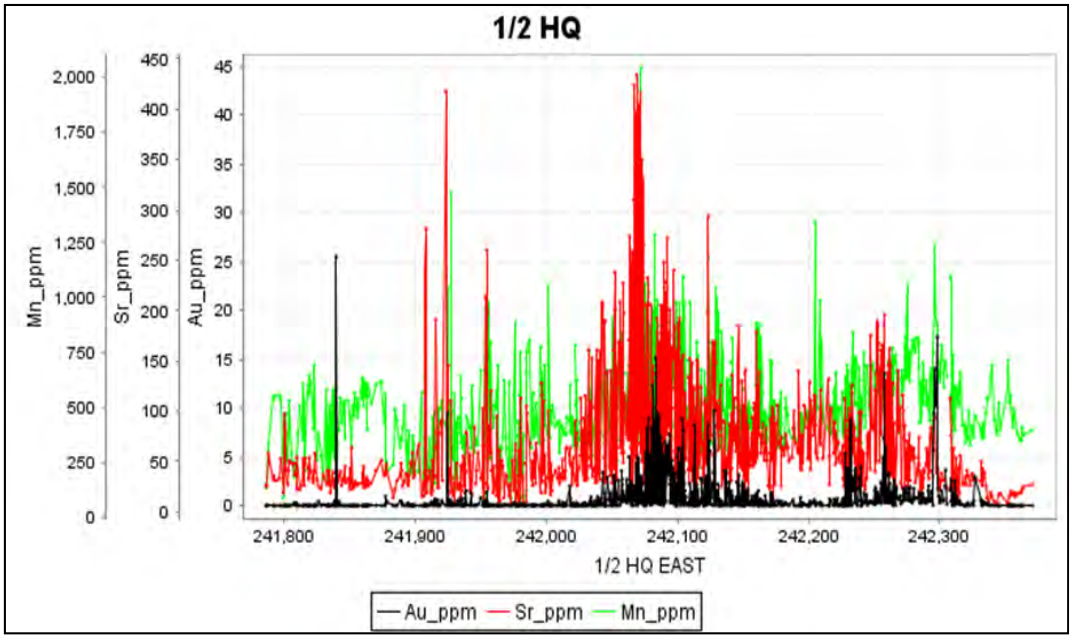
Moderately pervasive, dark green (Fe-rich) chlorite and calcite veinlets that have been strongly transposed by foliation characterize the outermost suite of alteration minerals at Fekola. Trace amounts of tourmaline are locally observed in strongly altered siltstone.

Multi-element analyses via inductively coupled plasma mass spectrometry (ICP-MS) and portable x-ray fluorescence (p-XRF) show weak, but positive correlations between Au and Mo, Cu, As, Co, Ni, Sr, Mg, Mn, Fe and S. Corresponding peaks in Sr, Mn and Au in an east-west profile across the zone suggest that elements may be tracking Au-related dolomite-ankerite alteration (Figure 7.4.2).

Figure 7.4.1 Veinlet (A) and Disseminated-style (B) Pyrite Mineralization in Fine Grained Siltstone / Mudstone of the Turbidite Unit

Veinlets exhibit tight F1 intrafolial folds indicating that the mineralization event at Fekola occurred early in the deformation history of the deposit

Figure 7.4.2 East-west Profile of ICP-MS Analyses from ½ HQ Diameter Drill Core (n=1110)



Coincident peaks in Sr and Mn with Au over main mineralized zone at 242,090 E suggest multi-element data can be used to map Au-related carbonate alteration.

FEKOLA GOLD PROJECT

NI 43-101 - DEPOSIT TYPES

3174-STY-002

Table of Contents

			Page
8.0	**DEPOSIT TYPES**		**8.1**
	8.1	Deposit Model	8.1
	8.2	Comparison of Fekola to Loulo Gold Deposits	8.2

TABLES

Table 8.2.1	Comparison: Loulo Mining Camp Deposits to Fekola Deposit	8.4

FIGURES

Figure 8.1.1	Orogenic Deposits	8.1

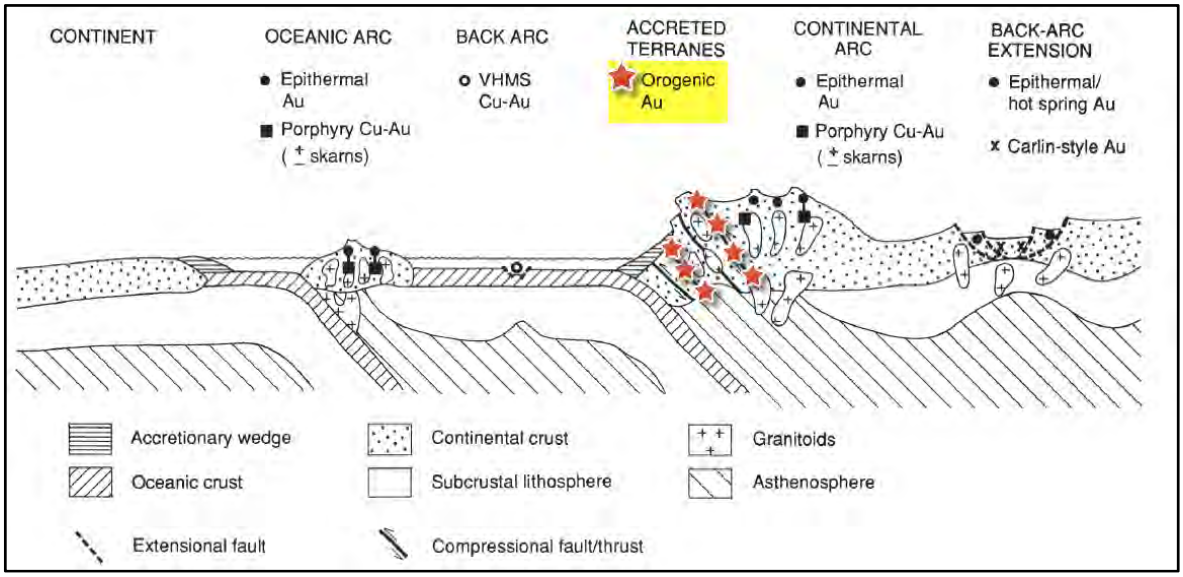
8.0 DEPOSIT TYPES

8.1 Deposit Model

The Fekola deposit is considered to be an example of an orogenic-style gold deposit. Global examples of this deposit type are located at Kalgoorlie (Australia), Val d'Or (Quebec) and Ashanti (Ghana).

Orogenic gold deposits occur in variably deformed metamorphic terranes formed during compressive or transpressional deformation events at convergent plate margins in collisional or accretionary orogens (Groves, et al, 2000) (Figure 8.1.1). Deposits are known from the Middle Achaean to younger Precambrian age, and throughout the Phanerozoic. These host geological environments are typically volcano-plutonic or clastic sedimentary terranes, but gold deposits can be hosted by any rock type. There is a consistent spatial and temporal association with granitoids of a variety of compositions. Host rocks are typically metamorphosed to greenschist facies, but locally can achieve amphibolite or granulite facies conditions.

Figure 8.1.1 Orogenic Deposits



Modified from Groves et al, 2000

Gold deposition occurs adjacent to first-order, major deformation zones, typically on the order of several hundreds of kilometres in strike, with a width of up to several hundreds of metres. First order deformation zones commonly display multiple, complex deformation events. Economic mineralization is typically hosted by second or third order structures, with jogs, changes in strike, or bifurcations of the first-order systems providing favourable locations for fluid focusing (Weinberg et al., 2004). Fault intersections, fold hinge zones and other areas of low mean stress may serve as areas favourable for mineralization.

Deposit-scale hydrothermal alteration typically displays a strong lateral zonation, the scale, intensity, and mineralogy of which are functions of wall rock composition and level of crustal

emplacement of the deposit. The main alteration minerals include carbonates (calcite, dolomite, and ankerite), sulphides (pyrite, pyrrhotite or arsenopyrite), alkali-rich silicate minerals (sericite, fuchsite, albite, and less commonly, K-feldspar, biotite, paragonite), chlorite, and quartz (Groves, et al, 1993).

Mineralization can be disseminated, or vein hosted, and displays a timing that is kinematically late with respect to the deformation of the host stratigraphy. Mineralization is also typically syn- to post-peak regional metamorphic conditions. Quartz is the primary constituent of veins, with lesser carbonate and sulphide minerals. Minor accessory minerals including albite, chlorite, white mica (fuchsite in ultramafic host rocks), tourmaline, and scheelite can accompany the veins. Carbonates include calcite, dolomite, and ankerite.

Gold in orogenic deposits is typically associated with sulphide minerals, including pyrite, pyrrhotite, chalcopyrite, galena, sphalerite, and arsenopyrite, but can locally occur as free gold. In volcano-plutonic settings, pyrite and pyrrhotite are the most common sulphide minerals in greenschist and amphibolite grade host rocks, respectively. Arsenopyrite can be the predominant sulphide mineral in mineralization hosted by sedimentary rocks. Gold to silver ratios typically range from 10:1 to less commonly, as little as 1:1. Most orogenic gold deposits contain <2% to 5% sulphide minerals and historically exhibit gold grades in the range of 5 to 30 g/t Au, although West African examples tend towards a lower 1 to 1.5 g/t Au mean grade, with local examples as high as approximately 10 g/t Au.

The mineralization/alteration footprints of some of the larger examples of orogenic deposits are in the order of 2 km, to much as 10 km long by approximately 1 km wide, and can persist to depths of over 1 to 2 km vertical extents.

8.2 Comparison of Fekola to Loulo Gold Deposits

The Fekola deposit is south of a major mineral camp, known as the Loulo mining camp, which has been interpreted to contain two distinct deposit types, termed Gara-style and Yalea-style. Based on Laurence et al. (2013) the key features of the two deposit styles are:

- Gara-style:
  - the deposits typically occupy a narrow zone (<2 km wide) along the border with the Senegal-Mali shear zone and include ankerite-rich shear vein stockworks and sulphide disseminations hosted in folded tourmalinized quartz wackes or as disseminated lodes along more discrete shear structures (e.g. Gounkoto and Yalea North)
  - these deposits are typified by intensely metasomatized wall rock consisting of albite and multistage tourmaline alteration. Mineralized zones are distinguished by Fe-rich sulphide assemblages (pyrite dominant), and polymetallic minor and trace phases including elevated concentrations of P, LREE, Ni, Cu, and Co

- Yalea-style:
  - the deposits are typically quartz ± carbonate vein lodes or sulphide stringer zones hosted in altered (tourmaline-absent) brittle-ductile shears that developed approximately 8 km to the east of the Senegal-Mali shear zone
  - the mineralization is associated with high As values, with the As mainly occurring in the form of arsenopyrite and arsenian pyrite. Deposits typically contain extremely low base and rare earth metal concentrations. Wall-rock alteration is characterized by carbonate-silica-chlorite-sericite ± albite assemblages.

In comparison to these styles, Dorling (2013) notes the following for the Fekola mineralization:

- At Fekola, gold mineralization is predominantly disseminated in nature, associated with an envelope of intensely metasomatized wall rock consisting of dolomite and albite with only traces of tourmaline.
- Gold at Fekola is associated with pyrite and lacks the Yalea-style association between gold mineralisation and arsenian pyrite, arsenopyrite and other base metals.
- Fekola alteration is dominated by dolomite, whereas the Loulo deposits are characterised by ankerite.

The similarities between the three deposit types include:

- The strong sodic alteration observed at each deposit.
- Prevailing greenschist-facies metamorphism at each deposit.
- Metasedimentary host rocks at each deposit.
- Located on the eastern side of the main controlling structure, the Senegal-Mali Shear Zone.

Table 8.2.1 Comparison: Loulo Mining Camp Deposits to Fekola Deposit

Characteristics	Gara-style Deposits	Yalea-style Deposits	Fekola-style Deposits
Examples	Gara, Gounkoto, Yalea North, P-129, P-64	Yalea Main, P-125, Loulo-3, Baboto, Faraba	Fekola Main
Mineralogy			
Mineralization styles	Ankerite–quartz vein-hosted, breccia-hosted and disseminated styles	Quartz vein-hosted, quartz–carbonate vein hosted, disseminated styles	disseminated style; local pyrite veinlets
Major sulphides	Pyrite (> 90%), Ni and Co substitution	Arsenian pyrite and arsenopyrite (> 90%)	Pyrite
Minor / trace sulphides	Chalcopyrite, gersdorffite, pentlandite, arsenopyrite, tennantite, cobaltite, clausthalite, galena, sphalerite. Also Ni-bearing phases and molybdenite	Pyrrhotite + chalcopyrite + tennantite ± tetrahedrite ± galena (trace) ± jamesonite ± bournonite	Chalcopyrite + tennantite ± tetrahedrite ± galena (trace)
Other minerals	Abundant monazite and scheelite, less commonly xenotime and magnetite	Trace scheelite and magnetite, limited or no appearance of REE phosphates	Trace scheelite and magnetite
Metal association	Fe–Cu–REE–P–W–As–Au ± Co–Pb–Se–Zn–Mo	Fe–As–Cu–Au–Ag and accessory Pb–Sb–W	Au, Mo, Cu, As, Co, Ni, Sr, Mg, Mn, Fe and S
Gold sites	Gold mainly confined to pyrite in numerous sites; low amounts of refractory gold	Gold situated in a range of Fe–As–Cu sulphides, high amounts of lattice-bound refractory gold	
Alteration			
Alteration types	Tourmalinization and albitization	Phyllic and silica–carbonate ± albite	carbonate–sericite + quartz; silica–carbonate ± albite
Alteration assemblages	Tourmaline association is tourmaline + quartz + ankerite + siderite ± rhodochrosite ± biotite ± chlorite Albitization association is albite + quartz + ankerite + hematite	Phyllic association is sericite + chlorite + quartz Silica–carbonate ± albite association is ankerite + quartz ± albite (varying amounts) ± hematite ± pyrite	Carbonate–sericite + quartz Silica–carbonate ± albite association is dolomite + quartz ± albite (varying amounts) ± hematite ± pyrite

Modified from Dorling (2013), after Laurence et al. (2013)

FEKOLA GOLD PROJECT

NI 43-101 - EXPLORATION

3174-STY-002

Table of Contents

			Page
9.0	**EXPLORATION**		**9.1**
	9.1	Grids and Surveys	9.1
	9.2	Geological Mapping	9.1
	9.3	Geochemical Sampling	9.2
	9.4	Geophysics	9.5
	9.5	Pits and Trenches	9.8
	9.6	Petrology, Mineralogy and Research Studies	9.8
		9.6.1 Schandl (2012)	9.8
		9.6.2 Pathfinder (2012 to 2013)	9.9
		9.6.3 Panterra Geoservices (2014)	9.9
	9.7	Exploration Potential	9.9
		9.7.1 Fekola Deposit	9.9
		9.7.2 Exploration Targets	9.9
	9.8	Comments on Section 9	9.12

TABLES

Table 9.3.1	Summary Table, Geochemical Sampling, Médinandi Exploitation License	9.3
Table 9.4.1	Geophysical Surveys	9.5

FIGURES

Figure 9.2.1	Interpreted Geological Map from Geophysics and Drilling	9.2
Figure 9.3.1	Geochemical Anomaly Plan, Médinandi Exploitation License	9.4
Figure 9.4.1	Metal Factor from Gradient Array IP	9.6
Figure 9.4.2	Airborne EM Response (142 ms)	9.7
Figure 9.7.1	Exploration Potential (Drill Hole Grade Thickness)	9.11

9.0 EXPLORATION

9.1 Grids and Surveys

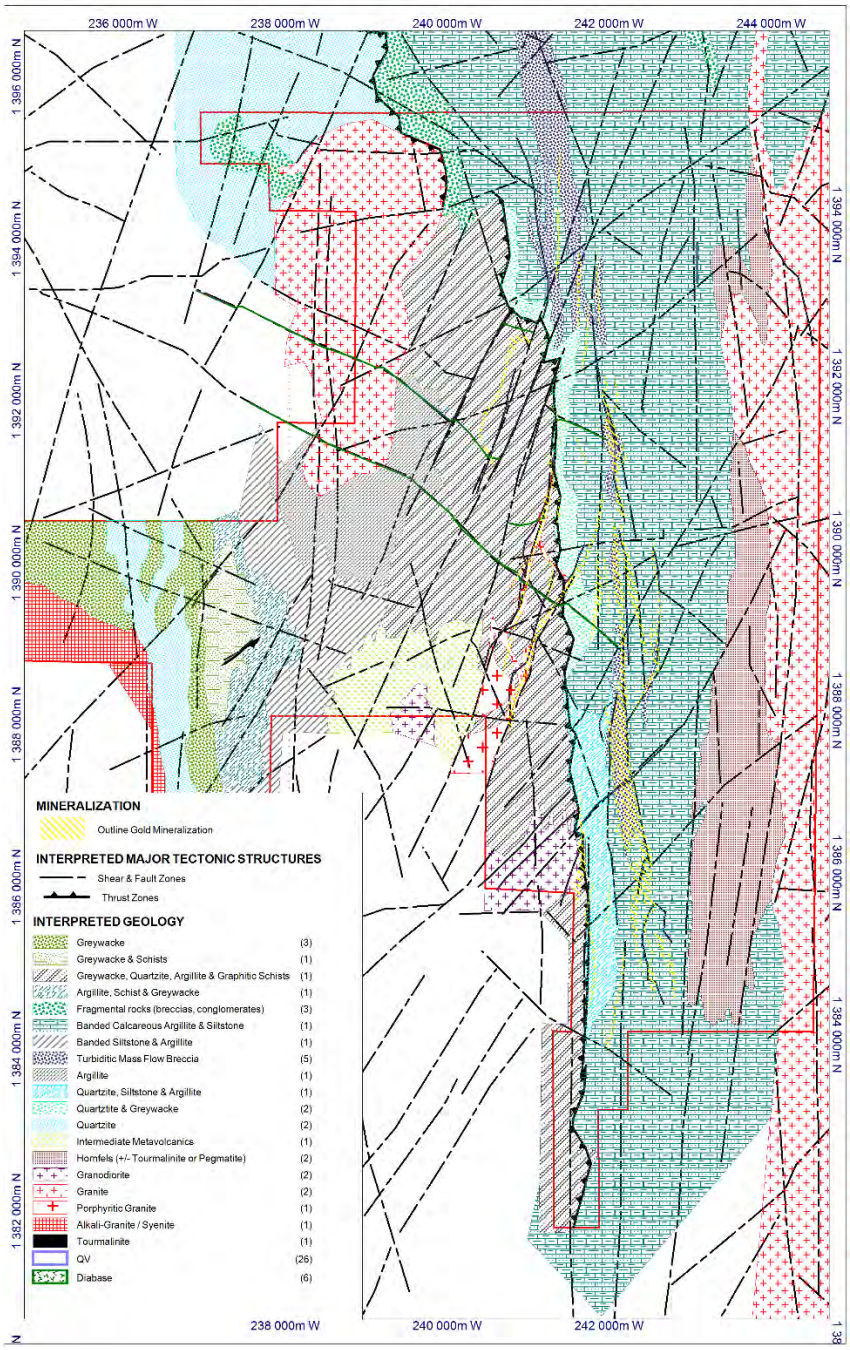
A light detection and ranging (LiDAR) survey was undertaken in 2012. The projection utilised for the whole project was UTM 29N, WGS84. The EGM2008 Geoid Model was used to transform from ellipsoid heights to orthometric heights (metres above sea level). The survey has a contour accuracy of ±0.5 m.

All drill hole locations are surveyed with a Leica 1230 DGPS.

9.2 Geological Mapping

Geological mapping is of limited use in the Project area, due to the minimal amount of outcrop, and the deep regolith cover. Regolith mapping was undertaken to identify which portions of the tenure are covered by a lateritized in situ profile, as opposed to transported materials and alluvium, which would allow reliable soil sampling. A recent compilation using a combination of drilling and geophysics is presented in Figure 9.2.1.

Figure 9.2.1 Interpreted Geological Map from Geophysics and Drilling



9.3 Geochemical Sampling

Soil geochemistry has proven to be an effective exploration tool in the search for gold mineralisation in some areas where there are residual soils. This has broadly outlined anomalous areas that have been followed up. In areas of more complex regolith, particularly where there are transported laterite, alluvial gravels and silt, the interpretation of soil geochemistry results can be ambiguous due to masking or suppressing potentially anomalous areas. For example, the Fekola

deposit is largely blind to soil geochemistry due to a substantial portion of the deposit being covered by transported palaeo-channel gravels and laterite.

In the early Papillon programs, soil samples were analysed for gold only. Recent geochemical work suggests that gold mineralisation may be associated with elevated tungsten, copper and possibly arsenic values; however, the use of these pathfinder elements requires further evaluation.

Soil geochemical surveys have been completed in a number of phases with the initial focus around the Médinandi prospect and then extending out across the area of the Médinandi Exploitation License. The soil sampling surveys consisted of the collection of material from small pits that were typically excavated to 60 cm below surface on 80 m by 160 m spaced grid lines. Due to the regolith issues discussed above, B2Gold believe that a more effective method is to sample the top of saprolite and plan to use auger drilling for the upcoming exploration programs in order to obtain consistent and unambiguous samples.

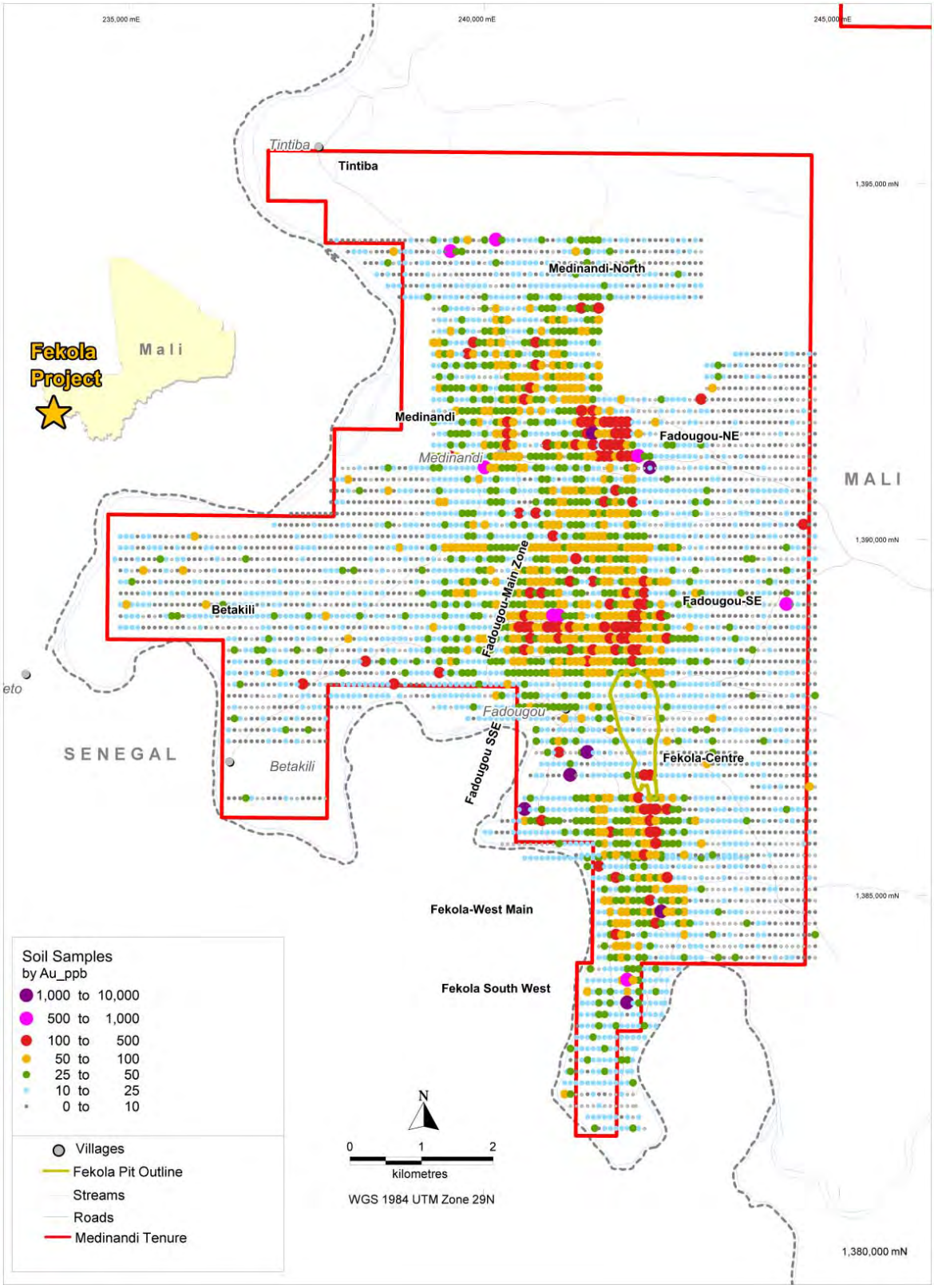
Termite mound and rock chip and grab sampling has also been performed within the Médinandi Exploitation License, delivering inconclusive results. The larger mounds with deeper extents are the preferred ones to sample.

The information in the B2Gold database for the Médinandi Exploitation License is summarized in Table 9.3.1. A compilation geochemical anomaly plan for gold for the exploitation license is included as Figure 9.3.1.

Table 9.3.1 Summary Table, Geochemical Sampling, Médinandi Exploitation License

Type	Count	Quality Control	Years Collected	Prospect / Area
Soil	7,490	215 duplicate samples	2007, 2008, 2010, 2011, 2012	Betakili Fadougou-NE Médinandi Médinandi-E Médinandi-N
Termite mound	285	15 duplicate samples	2010	Médinandi
Rock Grab	83		2013	Médinandi Tintiba-Sud Fadougou-SE Betakili

Figure 9.3.1 Geochemical Anomaly Plan, Médinandi Exploitation License



9.4 Geophysics

A number of phases of geophysical surveys have been completed over the deposit and the exploitation license area, including in 2007, 2008, 2010, 2011, 2012, and 2013 (Table 9.4.1).

The data have been used to develop the broad lithological and structural framework for the project area; however, no direct and distinct signature for the Fekola deposit is currently recognised within any of the geophysical datasets. The most useful dataset acquired is the gradient array IP data (Figure 9.4.1), which provides a good lithological contrast between units, as well as resolving project-scale structure.

The general lack of magnetite and low contrast between lithologies within the system reduces the efficacy of the airborne magnetic data (Figure 9.4.2). In addition, a lack of conductors within the lithological package hosting Fekola mineralisation has limited the effectiveness of the broad bandwidth EM system used.

Table 9.4.1 Geophysical Surveys

Year	Survey Type	Contractor	Comment
2007	Ground induced polarization (IP) and high resolution IP (HRIP)	Terratec	66 line km of data collected
2008	Helicopter-borne aeromagnetic	Geotech Airborne	Covered entire permit area. Line spacings of 160 m; total 845line km. Designed to test the continuity of the Fadougou Main Zone structure; identify any similar structures running parallel to the zone
2008	Ground IP/gradient	Sagax–Africa	47 lines on a grid of 160 m x 20 m; total of 94 line km
2010	Ground survey (type not specified	Sagax–Africa	162.9 line km over Fadougou, Fekola, Tintiba and Betakili
2011	Ground resistivity	Sagax–Africa	17.3 line km in support of evaluation of underground potable water sources
2011	Ground pole–dipole surveys	Sagax–Africa	17 profiles over 1600 m of strike, with lines spaced at 160 m intervals.
2012	Ground pole–dipole gradient-array	Sagax–Africa	69 lines situated northwest of Médinandi, for a total of 9,360 m.
2013	Ground gravimetric	Atlas Geophysics	Trial study to determine effectiveness of technique in the northern portion of the Fekola deposit

Figure 9.4.1 Metal Factor from Gradient Array IP

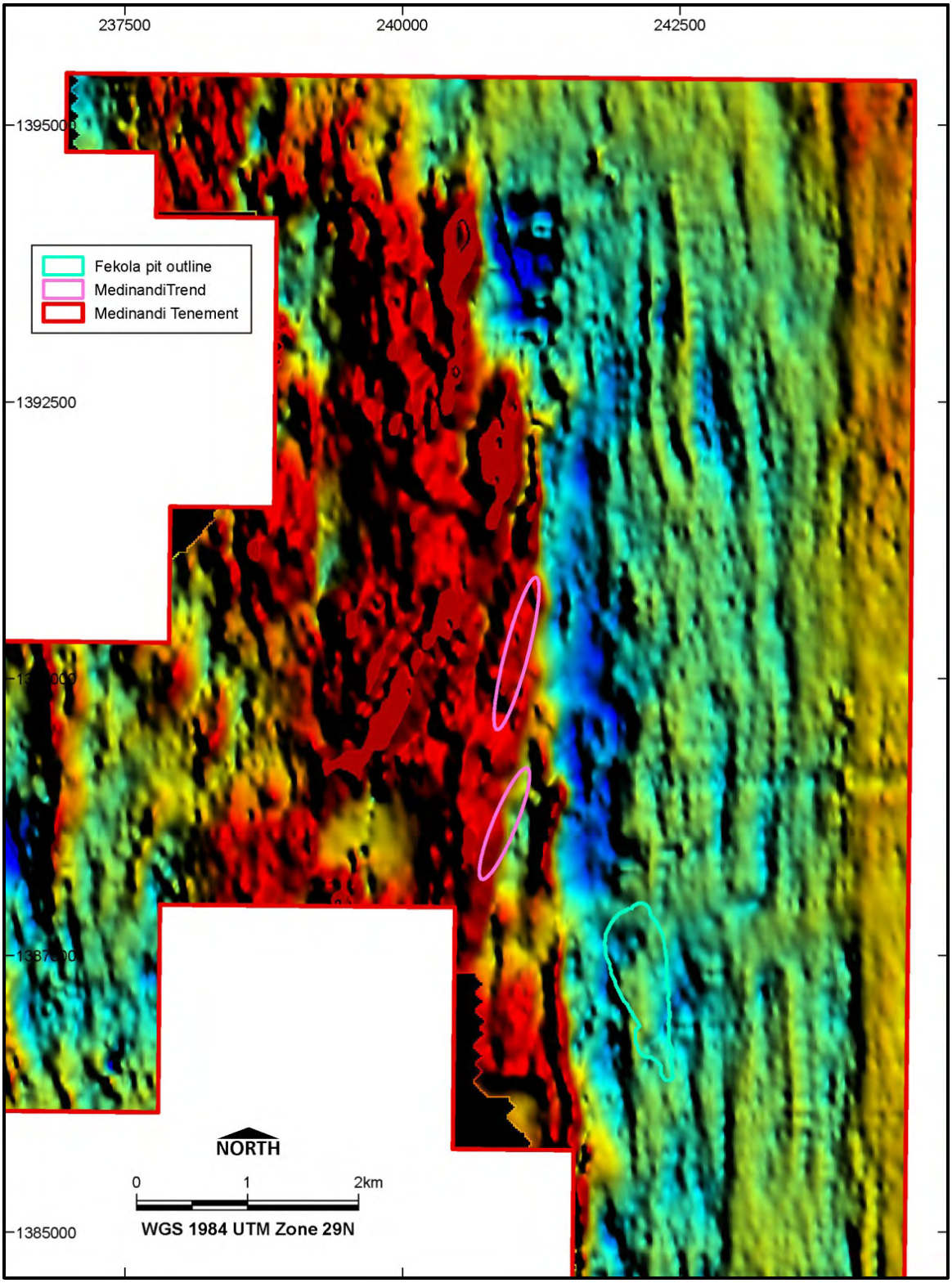


Figure prepared by Papillon, 2013

Figure 9.4.2 Airborne EM Response (142 ms)

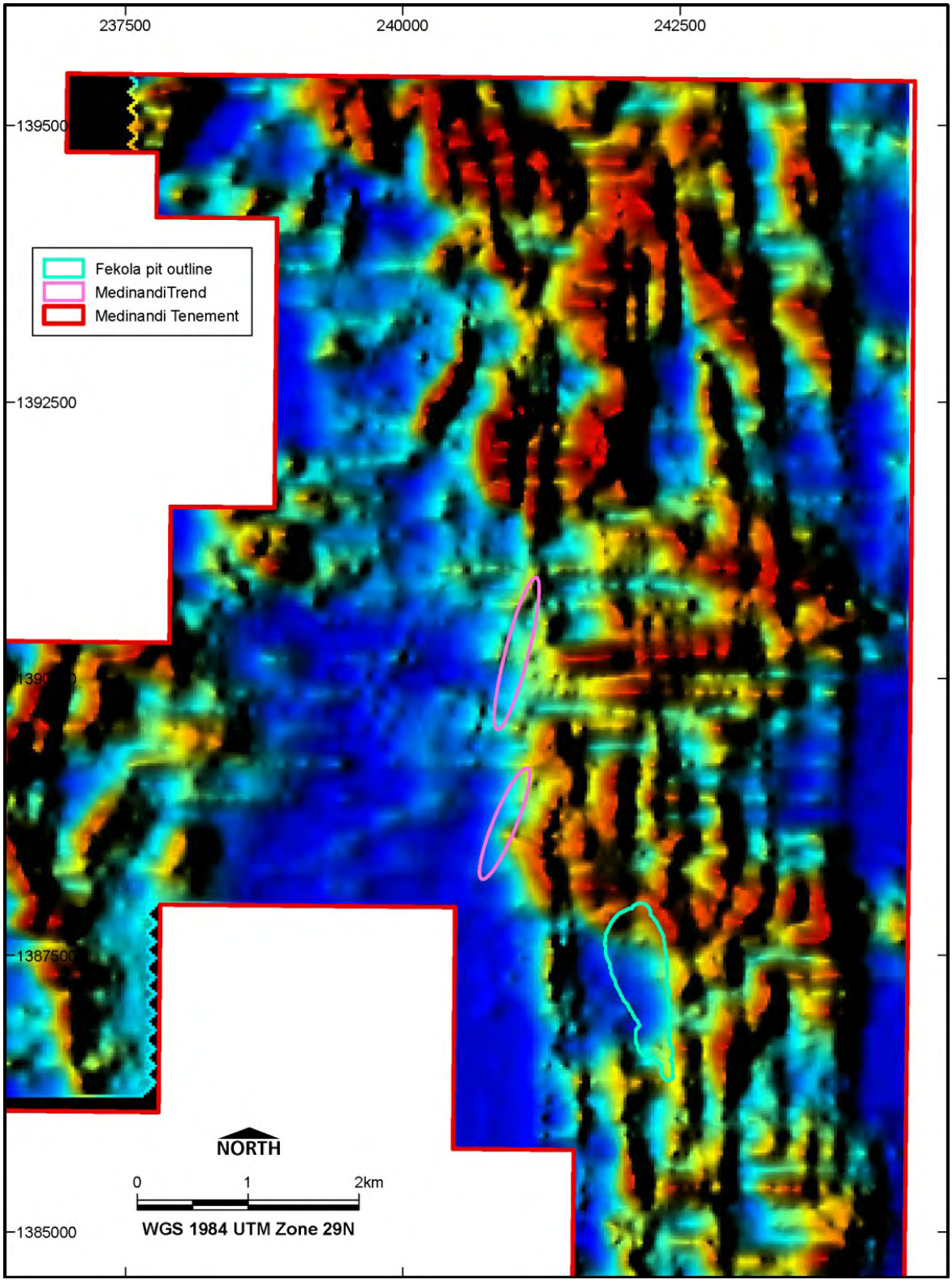


Figure prepared by Papillon, 2013

9.5 Pits and Trenches

Some pitting and trenching was performed over the exploitation license during the legacy campaigns. At Fadougou East, there were six trenches for 123 m excavated in 2008 and one trench, 150 m in length, was excavated by Songhoi in June 2010, at Fadougou Northwest.

A total of 48 pits were excavated in 2013 as part of the geotechnical appraisal of the planned plant and tailings storage facility (TSF) area and a further 62 geotechnical test pits were excavated in the same area during Q1 2014.

9.6 Petrology, Mineralogy and Research Studies

9.6.1 Schandl (2012)

In 2012, prior to much of the infill drilling on the Fekola deposit, Dr. Eva Schandl described a suite of 40 thin sections from the deposit to provide information on the lithologies, mineralogy, and hydrothermal alteration. Results of the work included:

- The evaluation identified four main sedimentary units: schists, sandstone / siltstone, conglomerates and breccias, and arkose. One sample was tentatively identified as having a mafic protolith.
- The sediment samples showed brecciation, fracturing and hydrothermal alteration. Secondary minerals such as quartz, carbonate, sericite, biotite, and anhydrite were noted to replace the earlier minerals by fracture-filling and cross-cutting veins, as well as via dissolution and precipitation.
- The mineralogy of the sediments was considered to be relatively simple, consisting predominantly of carbonate and quartz (hydrothermal and detrital). Other minerals observed included biotite, muscovite, sericite and chlorite. Secondary minerals, in lesser quantities, consisted of anhydrite, tourmaline, serpentine, and scapolite.
- Early silicification is represented by the replacement of sedimentary carbonates by chalcedony, and late silicification, by the partial replacement of late carbonate veins by aggregates by fine-grained chalcedony.
- The average carbonate content of the Fekola sediments is 60% and some rocks contain 80 to 90% carbonates. Secondary carbonates occur mostly as veins and as fracture-filling aggregates.
- Very fine-grained biotite and sericite are abundant in the schists, where they define the rock fabric. Both frequently occur as anastomosing veins that boudinage the quartz and carbonates. In some rocks, where biotite-rich laminae were part of the original sediments, the laminae are now fragmented and the fractures are filled by fine-grained carbonate.

Pyrite is the dominant sulfide in the rocks, with a concentration that ranges from 0 to 6%. The relatively small grains are most abundant in the siliceous calcareous rocks and breccias, where they appear to have been part of the original sediments. Although some mobilization was noted, several of the pyrite are fragmented and have embayed, sutured grain boundaries, suggesting partial dissolution during hydrothermal alteration.

9.6.2 Pathfinder (2012 to 2013)

Pathfinder Exploration undertook a number of detailed petrographic analyses on thin section materials, corroborated with PIMA analyses on selected samples. Descriptions were provided of the major constituents of each sample, and a suggested protolith described.

9.6.3 Panterra Geoservices (2014)

Panterra examined 11 mineralised samples from Metallurgical drill holes drilled at Fekola.

Six of the samples were from a possible mudstone-siltstone protolith. The rest included a fine grained turbidite, a mylonite, one of a possible volcanic protolith, a possible conglomerate and a meta-pelite.

The rocks are overprinted with 62 to 86% carbonate and between 5 to 20% quartz content and dolomite alteration is carbonate of dolomitic or Mg-Fe composition.

A precursor albite + carbonate alteration is overprinted by more intense carbonate.

Pyrite is closely associated with intense carbonate alteration.

9.7 Exploration Potential

9.7.1 Fekola Deposit

The Fekola deposit remains open at depth. Additional areas which will warrant future exploration effort include:

- Down plunge of the main high grade zone where a step out hole (FKD_148, drilled by B2Gold), intersected significant mineralization where expected.
- Mineralised structures in the footwall and hanging wall of the main mineralization.
- Potential for new plunging high grade zones along strike of the main high grade zone where narrower intersections have been intersected at shallow depths.

9.7.2 Exploration Targets

Figure 9.3.1 shows the gold anomalism identified through geochemical sampling. Figure 9.7.1 shows the results of exploration drilling, presented as grade–thickness contours. Both plots indicate areas of gold anomalism that may be able to support additional exploration focus.

A further analysis of the data generated by previous work highlighted new and existing areas for further exploration. This exploration will include a 200 m x 200 m auger program to test the top of saprolite.

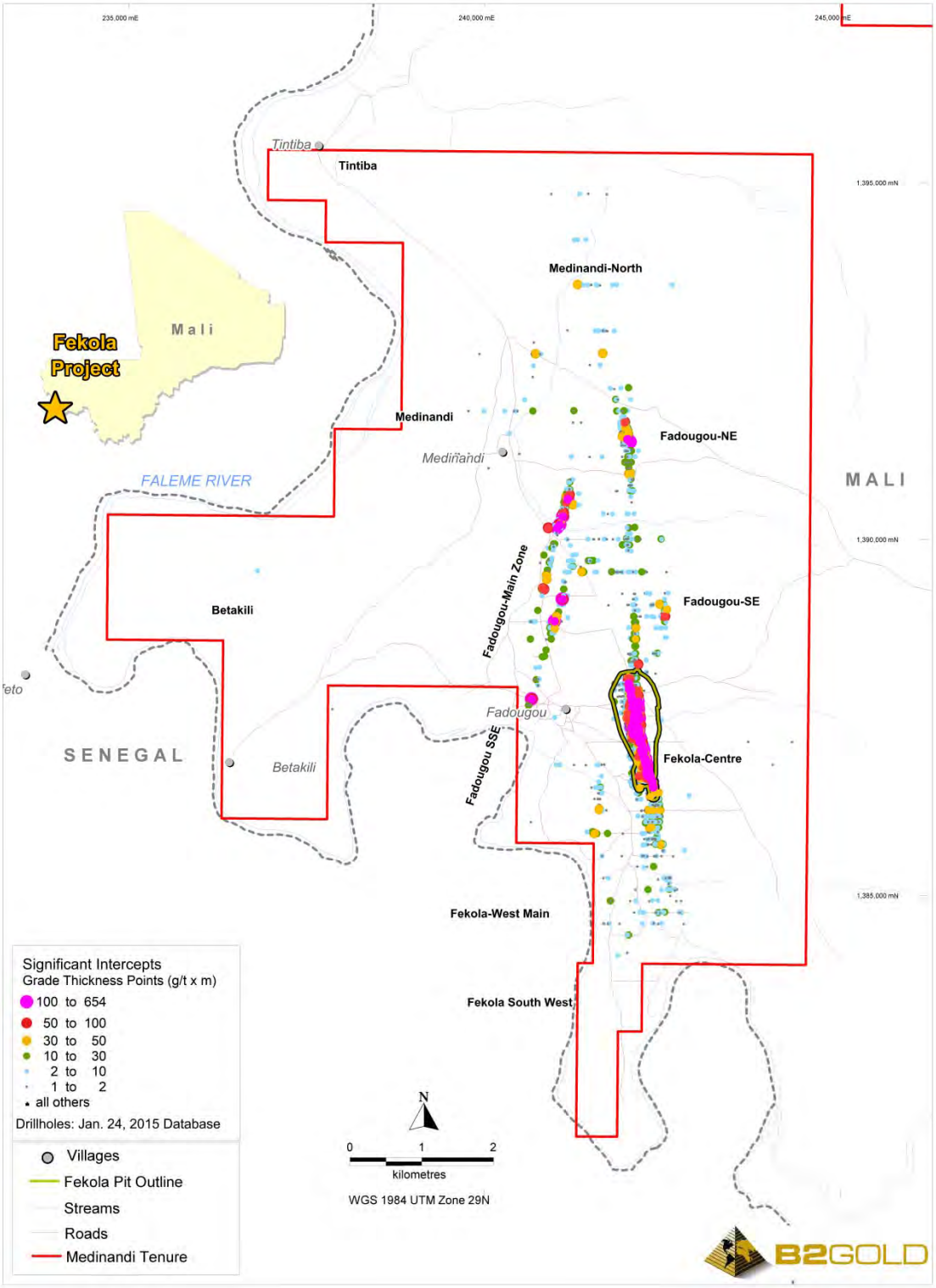
Fadougou NE prospect is along strike from the Fekola deposit and potentially represents an analog target to Fekola. Wide spaced and relatively shallow RC drilling in this area has confirmed anomalous gold intersections beneath laterite cover.

The Tintiba anomaly was identified as a structural target based on geophysical data. Two trenches, excavated 120 m apart, returned significantly elevated gold values. The trench anomalism is supported by elevated gold values returned from soil and pit sampling. An auger drilling program is scheduled this year to refine the target before follow up drilling is undertaken, if warranted.

The Médinandi prospect was initially delineated as a large gold-in-soil anomaly. However, follow-up work undertaken by Papillon suggests the anomaly may be an artifact of artisanal mining activity and not be related to a cohesive mineralized structure.

Betakili is a structural target derived from geophysical data interpretation.

Figure 9.7.1 Exploration Potential (Drill Hole Grade Thickness)



9.8 Comments on Section 9

In the opinion of the QP:

- The exploration completed to date has been appropriate, leading to the discovery of the Fekola deposit. The soil sampling was representative of some of the areas sampled, however in the future, auger drilling through the laterite and alluvial cover would aid in covering all the areas with geochemical sampling.
- The potential for extending the Fekola main deposit down plunge is good.
- A better understanding of the Fekola deposit, its location in a regional and local context with respect to lithologies, alteration, structure and regolith, will generate additional targets and enable their ranking for follow up.

FEKOLA GOLD PROJECT

NI 43-101 - DRILLING

3174-STY-002

Table of Contents

			Page
10.0	**DRILLING**		**10.1**
	10.1	Introduction	10.1
	10.2	Legacy Drilling	10.1
	10.3	Drill Data for January 2013 PEA Mineral Resource Estimate (as at 15 January 2013)	10.8
	10.4	Drill Data for August 2013 Mineral Resource Estimate Update (as at 5 August 2013)	10.10
	10.5	Drill Data for February 2015 Mineral Resource Estimate Update (as at January 24, 2015)	10.10
	10.6	Drill contractors	10.11
	10.7	Papillon / B2Gold Drill Methods	10.11
		10.7.1 Auger, Rotary Air Blast and Aircore	10.11
		10.7.2 Reverse Circulation	10.11
		10.7.3 Core Drilling	10.12
	10.8	Papillon/B2Gold Geological Logging	10.12
		10.8.2 RC	10.14
		10.8.3 Core	10.15
		10.8.4 Exploration-stage Geotechnical Logging	10.17
		10.8.5 Magnetic Susceptibility	10.17
	10.9	Recovery	10.17
	10.10	Collar Surveys	10.17
	10.11	Downhole Surveys	10.18
		10.11.1 RC	10.18
		10.11.2 Core	10.18
	10.12	Condemnation, Geotechnical and Hydrogeological Drilling	10.18
	10.13	Metallurgical Drilling	10.21
	10.14	Sample Length / True Thickness	10.21
	10.15	Drilling Since Completion of Resource Estimates	10.25
	10.16	Comments on Section 10	10.27

TABLES

Table 10.2.1	Drilling Summary, Médinandi Exploitation License, by Prospect Area	10.2
Table 10.2.2	Drilling Summary, Médinandi Exploitation License, by Year and Prospect Area (Current as at 24 January 2015)	10.3
Table 10.2.3	Drilling Summary, Médinandi Exploitation License, by Year (Current as at 24 January 2015)	10.5
Table 10.3.1	Summary of Fekola Drilling Database as at 15 January 2013 (Used for January 2013 PEA Mineral Resource)	10.8
Table 10.4.1	Summary of Fekola Drilling Database as at 5 August 2013 (Used for August 2013 Mineral Resource Estimate)	10.10
Table 10.5.1	Summary of Fekola Drilling Database as at 24 January 2015 (Used for February 2015 Mineral Resource Estimate)	10.11
Table 10.8.1	Geological Logging Codes	10.13
Table 10.14.1	Selected Drill Intercept Table	10.22

FIGURES

Figure 10.2.1	Drill Collar Location Plan, Médinandi Exploitation License	10.6
Figure 10.2.2	Fekola Deposit, Drill Collar Location Plan	10.7

Table of Contents (Continued)

Page

Figure 10.3.1	Fekola Deposit, Comparison Drill Collar Location Plan as at January, 2013 (Jan 2013 Model), August 2013 (Aug 2013 Model) and January 2015 (Feb 2015 Model)	10.9
Figure 10.8.1	Example Wet / Dry Chip Sample Recording	10.15
Figure 10.8.2	Example Wet / Dry Core Record	10.16
Figure 10.12.1	Location Plan, Geotechnical Drilling	10.20
Figure 10.15.1	Drilling Post 24 January 2015	10.26

10.0 DRILLING

10.1 Introduction

Some drilling was done on the Médinandi Exploitation License before 2007, previous to Papillon's ownership (refer to Section 6). The historic drilling and drill sampling are not considered part of the current active database, and are not used in support of the Mineral Resource estimates.

The working drill database, current as of 24 January 2015, focuses on drilling undertaken by B2Gold, Papillon and its predecessor companies from January 2007 to January 2015. Within the database are a total of 1,261 RC drillholes (145,287 m), 62 holes that commenced with an RC collar but were completed with a core tail (RC–DD) drillholes (18,727.8 m), and 190 core drillholes (50,832.8 m).

In addition are included 25 Water Boreholes (2,115.1 m) which were drilled using RC methods plus 29 core holes (2,437.25 m) and 25 RC holes (2,001 m) that have been classified as Geotechnical holes.

Also RAB drilling (1,166 holes; 24,115 m), aircore drilling (300 holes, 11,257 m), and trenching (seven trenches, 273 m) were completed as part of early exploration efforts and are not used in support of Mineral Resource estimates.

The drill programs are summarized by prospect area in Table 10.2.1 and by year in Table 10.2.2; drill collar locations coloured by drill type are shown in Figure 10.2.1.

Drilling as of 24 January 2015 within the Fekola deposit area totals 76,228.9 m of RC (554 drillholes), 18,727.8 m of RC-DD (62 drillholes), and 47,820.6 m of core (170 drillholes). This drilling is shown in Figure 10.2.2.

10.2 Legacy Drilling

Very limited information is available on the logging and surveying practices used prior to 2010. Colonial Resources (2010) notes that drillhole collars were surveyed using a hand held global positioning instrument (GPS).

Table 10.2.1 Drilling Summary, Médinandi Exploitation License, by Prospect Area

Prospect	Total Holes RAB	RAB (m)	Total Holes AC	AC (m)	Total Holes RC	RC (m)	Total Holes RC-DD	RC-DD (m)	Total Holes DDH	DDH (m)	Grand Total	Grand Total
Betakili	251	3,930	191	7,066	-	-	-	-	-	-	442	10,996
Fadougou-E	33	448	-	-	63	8,435	-	-	-	-	96	8,883
Fadougou-MZ	-	-	-	-	181	19,285	-	-	23	3,301	204	22,586
Fadougou-NE	278	6,332	-	-	119	16,556	-	-	-	-	397	22,888
Fdaougou-SE	430	9,888	109	4,191	149	21,639.1	-	-	4	1,208.3	692	36,926.4
Fekola Centre	174	3,517	-	-	437	55,872.9	62	18,727.8	170	47,820.6	858	126,533.95
Fekola-W Main	84	1,557	-	-	80	10,453	-	-	-	-	80	10,453
Medinandi-N	-	-	-	-	273	16,506.1	-	-	2	345	275	16,851.1
Tintiba-Nord	-	-	-	-	9	656	-	-	-	-	9	656
Total	**1,166**	**24,115**	**300**	**11,257**	**1,311**	**149,403.1**	**62**	**18,727.8**	**219**	**53,270.55**	**3,053**	**256,773.45**

Note: RC = reverse circulation; RC-DD is RC pre-collar and diamond core tail; RAB = rotary air blast, AC = aircore.
Hydrology holes and some Geotechnical holes are included in RC drilling and the remaining Geotechnical holes are included in DDH drilling
Since the last August 2013 update there has been a re organisation of geographical limits of drilling area which has caused some discrepancy in the amounts per prospect area.

Table 10.2.2 Drilling Summary, Médinandi Exploitation License, by Year and Prospect Area (Current as at 24 January 2015)

Year	Area Drilled	Total Holes RAB	RAB (m)	Total Holes AC	AC (m)	Total Holes RC	RC (m)	Total Holes RC-DD	RC-DD (m)	Total Holes DDH	DDH (m)	WBH (Water) RC	Total (m)	Geotech (DDH&RC)	Total (m)	Grand Total (m)	Grand Total
2007	Fadougou MZ					40	3,610			23	3,301					63	6,911
2007	Medinandi N					176	5,319			2	345	1	96			179	5,760
2008	Fadougou MZ					74	8,826									74	8,826
2008	Fadougou SE					13	1,548									13	1,548
2008	Fadougou E					9	1,170									9	1,170
2010	Fadougou MZ					45	4,334									45	4,334
2010	Fadougou NE					104	14,584									104	14,584
2010	Fadougou SE					15	1,879									15	1,879
2010	Fekola Centre	122	3,067			42	4,688									164	7,755
2010	Tintiba Nord					9	656									9	656
2011	Betakili	251	3,930	62	1,782											313	5,712
2011	Fadougou MZ					22	2,515									22	2,515
2011	Fadougou E					9	1,260									9	1,260
2011	Fadougou SE	463	10,336			10	1,412									473	11,748
2011	Fadougou NE	278	6,332			6	720									284	7,052
2011	Fekola Centre	52	450			141	17,446			8	1,984.6					201	19,880.6
2011	Fekola Sth					12	1,632									12	1,632
2012	Betakili			129	5,284											129	5,284
2012	Fadougou SE			109	4,191	35	5220									144	9,411
2012	Fekola Centre					105	11,690	42	11,877.1	126	35,281.7	2	202			275	59,050.8
2013	Fadougou E					29	4,455									29	4,455
2013	Fadougou NE					7	982									7	982
2013	Fadougou SE					74	11,420			4	1,208.3	2	160.1			80	12,788.4

Year	Area Drilled	Total Holes RAB	RAB (m)	Total Holes AC	AC (m)	Total Holes RC	RC (m)	Total Holes RC-DD	RC-DD (m)	Total Holes DDH	DDH (m)	WBH (Water) RC	Total (m)	Geotech (DDH&RC)	Total (m)	Grand Total (m)	Grand Total
2013	Fekola Centre					107	17,866	20	6,850.7	19	6,698.8	7	524.9	20	595.65	173	32,536.05
2013	Fekola South					30	5,446					2	174			32	5620
2013	Medinandi N					69	9,661					8	467.1	17	681	94	10,809.1
2014	Fadougou E					16	1,550									16	1,550
2014	Fadougou NE					2	270									2	270
2014	Fekola Centre					22	1,645			6	1,741.4	3	491	17	3,162.1	48	7,039.5
2014	Fekola South					36	3,201									36	3201
2014	Medinandi N					2	282									2	282
2015	Fekola Centre									2	272					2	272
Total		**1,166**	**24,115**	**300**	**11,257**	**1,286**	**147,288**	**62**	**18,727.8**	**190**	**50,832.8**	**25**	**2,115.1**	**54**	**4,438.75**	**3,058**	**256,773.45**

Note: RC = reverse circulation; RC-DD is RC pre-collar and diamond core tail; RAB = rotary air blast, AC = aircore

Table 10.2.3 Drilling Summary, Médinandi Exploitation License, by Year (Current as at 24 January 2015)

Year	Total Holes	Total Metres
2007	242	12671
2008	87	10374
2010	346	30378
2011	1314	49799.6
2012	548	73745.8
2013	415	67190.55
2014	104	12342.5
2015	2	272
Grand Total (2007 to 2015)	**3,058**	**256,773.45**

Figure 10.2.1 Drill Collar Location Plan, Médinandi Exploitation License

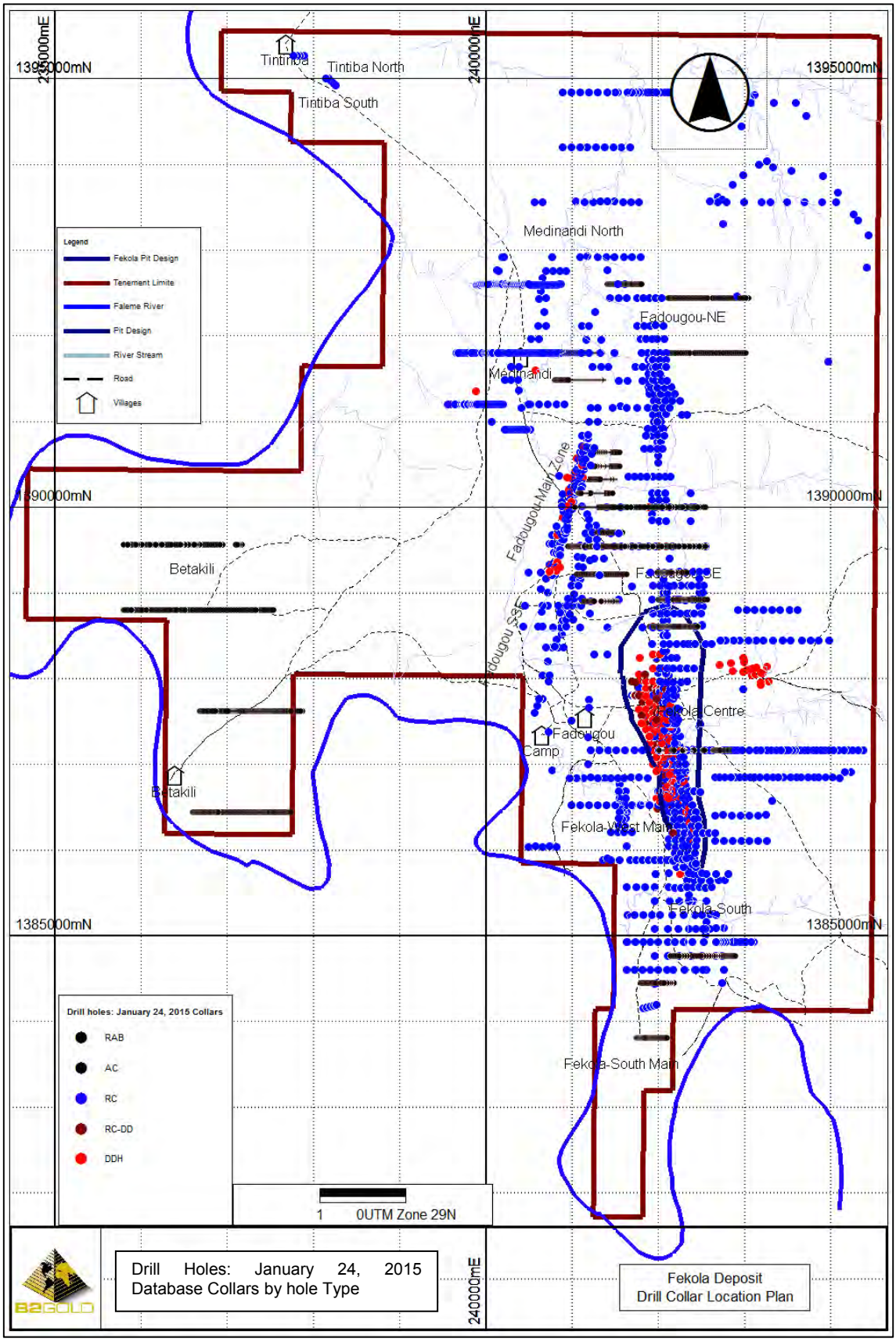
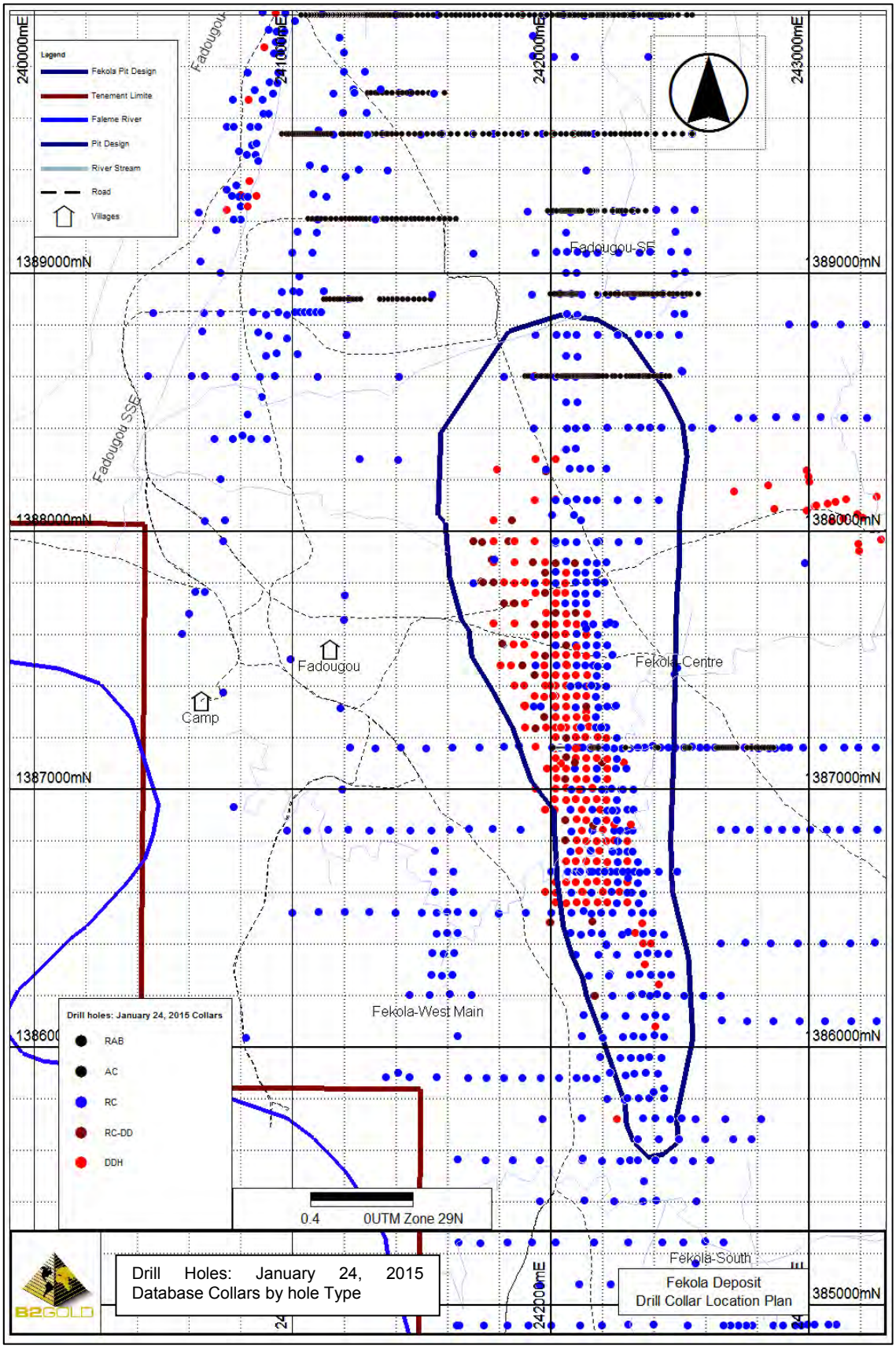


Figure 10.2.2 Fekola Deposit, Drill Collar Location Plan



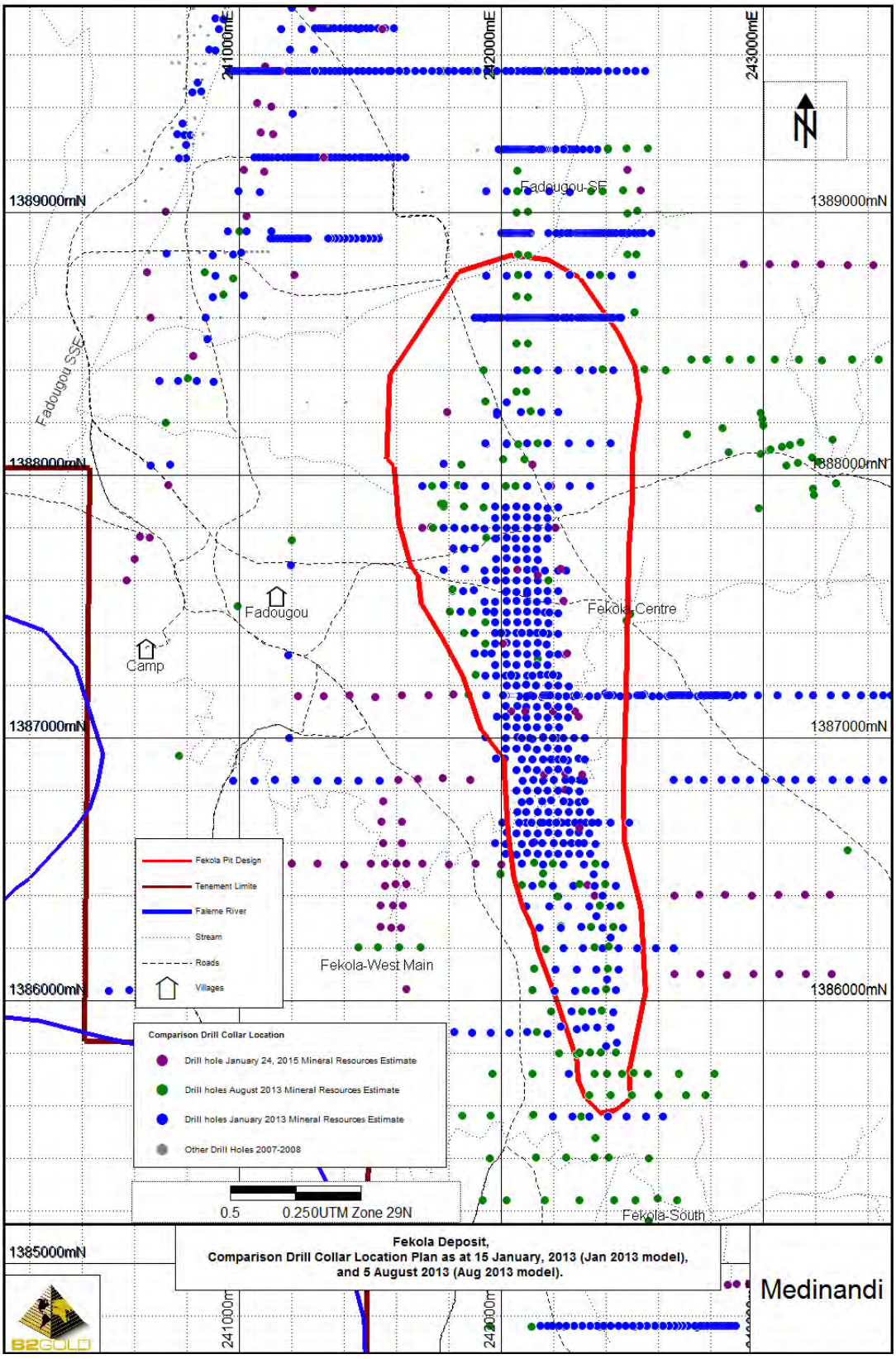
10.3 Drill Data for January 2013 PEA Mineral Resource Estimate (as at 15 January 2013)

A total of 428 drillholes (80,969 m) was available at the database cut-off date of 15 January 2013 that was used for the January 2013 PEA mineral resource estimate. Only RC, core and core with RC pre-collars were used in support of the January 2013 PEA Mineral Resource estimate. A summary of the number of holes and meterage by drilling type used in the January 2013 PEA Mineral Resource estimate is shown in Table 10.3.1 and drill collars are shown on Figure 10.3.1.

Table 10.3.1 Summary of Fekola Drilling Database as at 15 January 2013 (Used for January 2013 PEA Mineral Resource)

Drill Method	Hole Prefix	Number of Holes	Average Depth (m)	Total (m)	% of Total Metres
RC	FKCR, FSER, FKWM and FKC	255	130	33,022	41
RCD	FKRD	40	279	11,161	14
DDH	FKD	133	277	36,786	45
Totals		**428**	**189.**	**80,969**	**100**

Figure 10.3.1 Fekola Deposit, Comparison Drill Collar Location Plan as at January, 2013 (Jan 2013 Model), August 2013 (Aug 2013 Model) and January 2015 (Feb 2015 Model)



10.4 Drill Data for August 2013 Mineral Resource Estimate Update (as at 5 August 2013)

A total of 643 drillholes (127,148 m) was available at the data cut-off date of 5 August 2013 for the August 2013 Mineral Resource estimate update. As in the previous estimate, RC, core, and core with RC pre-collars were used in the estimate. A summary of the number of holes and meterage by drilling type used in the August 2013 update is shown in Table 10.4.1 and displayed on Figure 10.3.1.

An additional 215 drillholes were available at the time of the August 2013 Mineral Resource estimate update as compared to the January 2013 Mineral Resource estimate; however, only 56 of these holes (approximately 17,020 m) are within the immediate Fekola mineral resource estimate limits. The holes drilled close to the limits of the January 2013 resource supported the interpretations in the January 2013 PEA Mineral Resource estimate as to grade and geological continuity, and extended the mineralization down plunge.

Table 10.4.1 Summary of Fekola Drilling Database as at 5 August 2013 (Used for August 2013 Mineral Resource Estimate)

Drill Method	Hole Prefix	Number of Holes	Average Depth (m)	Total (m)	% of Total Metres
RC	FKCR, FSER, FKWM and FKC	439	147	64,663	51
RCD	FKRD	54	330	17,827	14
DDH	FKD	150	298	44,659	35
Totals		**643**	**198**	**127,148**	**100**

10.5 Drill Data for February 2015 Mineral Resource Estimate Update (as at January 24, 2015)

A total of 806 drillholes (143,372.98 m) were available at the data cut-off date of January 24, 2015 for the February 2015 Mineral Resource estimate update. As in the previous estimate, RC, core, and core with RC pre-collars were used in the estimate. Water boreholes and holes drilled for geotechnical purposes were also included. A summary of the number of holes and meterage by drilling type used in the February 2015 update is shown in Table 10.5.1 and displayed on Figure 10.3.1.

An additional 11 drillholes were available (after the 24 January cut-off date) at the time of the February 2015 Mineral Resource estimate update and all of these holes (322.1 m) are within the immediate Fekola mineral resource estimate limits. All holes recently drilled are infill on existing sections at 40 m x 40 m spacing for resource estimation purposes and drilled to confirm grade and geological continuity.

Table 10.5.1 Summary of Fekola Drilling Database as at 24 January 2015 (Used for February 2015 Mineral Resource Estimate)

Drill Method	Hole Prefix	Number of Holes	Average Depth (m)	Total (m)	% of Total Metres
RC	FKCR, FSER, FKWM, FKC, GTR & WBH	554	137.60	76,228.90	53.17
RCD	FKRD	62	302.06	18,727.80	13.12
DDH	FKD & GTD	170	281.30	47,820.60	33.49
Totals		**786**	**181.65**	**142,777.33**	**100**

10.6 Drill contractors

Colonial Resources (2010) indicated that the Randgold drilling was performed by West African Drilling Services (WADS), based out of Bamako. The majority of the drilling was on the Fadougou Main Zone and Fadougou NE.

Boart Longyear, Forage FTE drilling and African Mining Services (AMS) have provided drill rigs during the Papillon / Songhoi drill campaigns, with AMS currently the preferred contractor.

10.7 Papillon / B2Gold Drill Methods

10.7.1 Auger, Rotary Air Blast and Aircore

Exploration drilling has employed Auger, RAB and aircore methods as a first pass evaluation of gold-in-soil anomalies.

10.7.2 Reverse Circulation

RC drilling used face sampling hammer techniques rather than conventional RC methods where possible. Bit sizes during the 2012 to 2014 programs included 130 mm, 133 mm, and 137 mm with a 127 mm bit being used occasionally. The bit size used depended on the ground conditions and the progress of the hole. The sample weights at Fekola using these bit sizes are between 35 kg and 50 kg for a metre sample. A drill collar casing is typically used in the first 6 m of the drillhole to stop cave in and maintain good workable access to the drillhole.

Drill chips are collected at 1 m intervals during drilling and placed in chip trays. Trays are labelled with the prospect area, drillhole number and metre from–to interval.

During some programs at Fekola, significant water was encountered during drilling at approximately 40 to 60 m depth.

RC chips are stored in lidded, plastic core trays, most of which are kept in a shipping container / shed at site.

10.7.3 Core Drilling

Some of the core drilling for recent programs is performed using a RC collar with a core tail. Typically the change over occurs at depths of around 180m, but has been anywhere from between 40 m and 250 m, depending on depth to mineralisation.

Drilling difficulties can be encountered when penetrating the transported alluvial gravel that sits above the in situ weathered rock; this is normally countered by casing the hole with removable steel casing to just beyond this layer. Drillholes may also “lift” due to rock conditions at depth. There are also issues with “drifting” of holes which is mitigated by decreasing rotation speed. Downhole surveys are conducted every 30 m to monitor the dip and drift using a reflex (EZ-track) instrument.

Drill sizes include PQ (85 mm diameter), HQ (63.5 mm diameter) and NQ core (47.6 mm diameter). PQ core is typically used from the surface down to a few metres into hard rock in areas known to be clay rich, with paleochannel, and for geotechnical purposes. HQ3 core (triple tube method) is usually used in areas of broken ground for good recovery of core for structural orientations. HQ2 is used in areas of more competent ground.

Transportation of the core from the site to the sample yard is the geologist’s responsibility. Lids are used on core boxes if travelling a large distance or over rough ground.

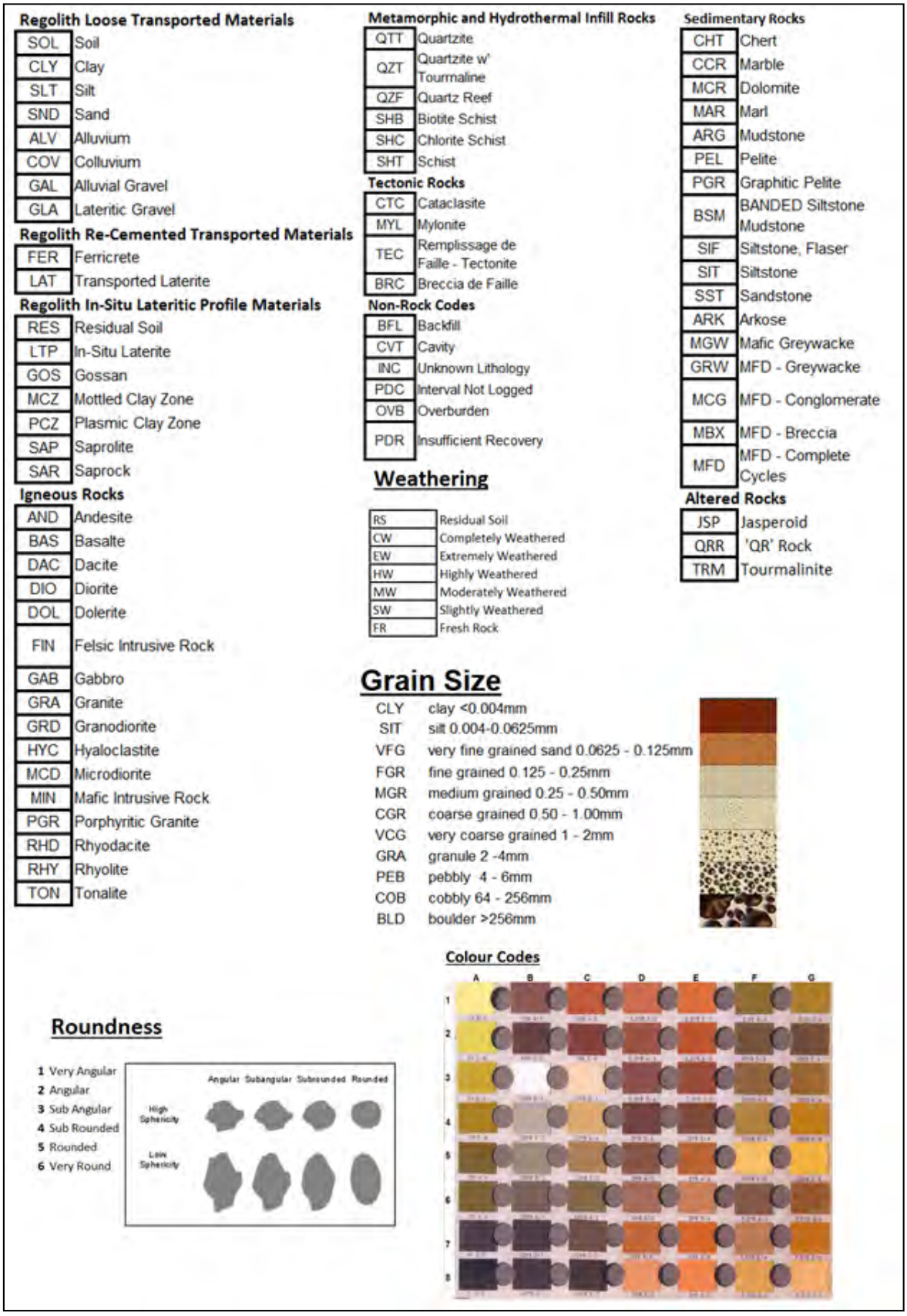
10.8 Papillon/B2Gold Geological Logging

Papillon developed a set geological legend for the Project area, and used this for trench, core, drill chip and surface mapping. The lithological rock types covered in the geological code include surface, sedimentary, igneous, and altered rock, metamorphic and tectonic lithologies. Logging definitions and standards were regularly reviewed for appropriateness for Fekola. The sampling by Papillon, previously was standardized at 1 m intervals. Recent sampling is also generally to 1 m intervals, but bounded by geological considerations with a minimum sampling width of samples >0.2 m by B2Gold.

Following an extensive review of the core and existing logging codes, B2Gold is in the process of enhancing the logging systems to incorporate the inclusion of all relevant geological data that will further improve the geological understanding of the deposit. This effort is ongoing and draws significantly on an assessment of the key parameters noted by Papillon geologists that have contributed to the geological model that informs the current Mineral Resource estimate.

Table 10.8.1 summarizes the types of lithological information collected.

Table 10.8.1 Geological Logging Codes



Regolith Loose Transported Materials

SOL	Soil
CLY	Clay
SLT	Silt
SND	Sand
ALV	Alluvium
COV	Colluvium
GAL	Alluvial Gravel
GLA	Lateritic Gravel

Regolith Re-Cemented Transported Materials

FER	Ferricrete
LAT	Transported Laterite

Regolith In-Situ Lateritic Profile Materials

RES	Residual Soil
LTP	In-Situ Laterite
GOS	Gossan
MCZ	Mottled Clay Zone
PCZ	Plasmic Clay Zone
SAP	Saprolite
SAR	Saprock

Igneous Rocks

AND	Andesite
BAS	Basalte
DAC	Dacite
DIO	Diorite
DOL	Dolerite
FIN	Felsic Intrusive Rock
GAB	Gabbro
GRA	Granite
GRD	Granodiorite
HYC	Hyaloclastite
MCD	Microdiorite
MIN	Mafic Intrusive Rock
PGR	Porphyritic Granite
RHD	Rhyodacite
RHY	Rhyolite
TON	Tonalite

Metamorphic and Hydrothermal Infill Rocks

QTT	Quartzite
QZT	Quartzite w' Tourmaline
QZF	Quartz Reef
SHB	Biotite Schist
SHC	Chlorite Schist
SHT	Schist

Tectonic Rocks

CTC	Cataclasite
MYL	Mylonite
TEC	Remplissage de Faille - Tectonite
BRC	Breccia de Faille

Non-Rock Codes

BFL	Backfill
CVT	Cavity
INC	Unknown Lithology
PDC	Interval Not Logged
OVB	Overburden
PDR	Insufficient Recovery

Sedimentary Rocks

CHT	Chert
CCR	Marble
MCR	Dolomite
MAR	Marl
ARG	Mudstone
PEL	Pelite
PGR	Graphitic Pelite
BSM	BANDED Siltstone Mudstone
SIF	Siltstone, Flaser
SIT	Siltstone
SST	Sandstone
ARK	Arkose
MGW	Mafic Greywacke
GRW	MFD - Greywacke
MCG	MFD - Conglomerate
MBX	MFD - Breccia
MFD	MFD - Complete Cycles

Altered Rocks

JSP	Jasperoid
QRR	'QR' Rock
TRM	Tourmalinite

Weathering

RS	Residual Soil
CW	Completely Weathered
EW	Extremely Weathered
HW	Highly Weathered
MW	Moderately Weathered
SW	Slightly Weathered
FR	Fresh Rock

Grain Size

CLY	clay <0.004mm
SIT	silt 0.004-0.0625mm
VFG	very fine grained sand 0.0625 - 0.125mm
FGR	fine grained 0.125 - 0.25mm
MGR	medium grained 0.25 - 0.50mm
CGR	coarse grained 0.50 - 1.00mm
VCG	very coarse grained 1 - 2mm
GRA	granule 2 -4mm
PEB	pebbly 4 - 6mm
COB	cobbly 64 - 256mm
BLD	boulder >256mm

Colour Codes

Roundness

1 Very Angular
2 Angular
3 Sub Angular
4 Sub Rounded
5 Rounded
6 Very Round

Angular Subangular Subrounded Rounded

High Sphericity

Low Sphericity

Figure prepared by Papillon, 2013

10.8.2 RC

RC drilling is rapid, with typically a minimum of 150 m achievable per shift, compared to 30 m to 40 m with Diamond Core.

At 180 m depth, typically a booster compressor is needed to go deeper for obtaining clean sample. In many cases, it is more cost effective and a better sample is obtained by continuing with Core from this depth.

Logging of RC chips at the drill rig is done at the chip tray and not from the chip boxes. The basic geological log includes:

- Primary lithology.
- Alteration.
- Mineralisation.
- Oxidation boundaries.
- Sample quality.
- Depth of water inflow (estimation of rates).
- Wet samples.
- Veining.
- Texture.
- Fabric.
- Presence of key minerals.
- Grain size (from grain size chart).

Areas of slow or hard drilling are marked onto the drill logs for geotechnical purposes.

Samples are placed onto the ‘sample table’ at the rig and photographed to provide a record of the general samples encountered. The table is photographed wet and dry (Figure 10.8.1) with the drillhole number and interval in metres displayed at the top of the chip sample table.

Figure 10.8.1 Example Wet / Dry Chip Sample Recording

Figure prepared Papillon, 2013

10.8.3 Core

Geological logging of core is performed in a similar manner to the RC described in Section 10.8.2. Particular attention is paid during logging to the following:

- Pyrite form and percentage.
- Alteration.
- Breccia units.

- Dark (mafic) sediments.
- Structures and Foliation.

At Fekola the core is marked up with orientation lines and structurally logged at the rig by a geologist, then brought back to the yard and labelled with sample numbers and cut lines drawn. Photography is done at the core yard under controlled 'indoor' conditions so that photographs are consistent in quality.

Core is typically photographed both wet and dry (Figure 10.8.2), prior to the core being sawed in half, sampled and bagged. For geotechnical cored holes the re-assembled runs are also photographed at the rig prior to transport.

Figure 10.8.2 Example Wet / Dry Core Record

Figure prepared by Papillon, 2013.

10.8.4 Exploration-stage Geotechnical Logging

Standard geotechnical logging on exploration and infill drill core collects information on fracture frequency and rock quality designation (RQD). Core is oriented for structural data collection, and both a goniometer and “rocket launcher” (spear) can be used. The structural geology logging sheet is used to record linear and planar structural features observed within the hole as either point or broad structural zones. The following features are recorded:

- Veining.
- Layering.
- Foliation.
- Faulting.
- Lithological contacts.
- Joints.
- Lineations.

10.8.5 Magnetic Susceptibility

The measuring of magnetic susceptibility (magsus) is one of the final stages of the logging process and takes place once the geological structural logging and sometimes even the core has been cut and sampled. The readings are taken every meter from start of hole to end of hole.

10.9 Recovery

The average core recovery is 97.2% for all drilling and is also 97.2% for holes completed within the Fekola deposit area. There does not appear to be a direct relationship between core recovery and gold grade for Fekola.

10.10 Collar Surveys

Drill collars for exploration drillholes are normally surveyed using a handheld GPS instrument.

In the deposit area, drillhole collars are picked up using a differential GPS, which has an accuracy of ± 10 cm.

There is a base station on the Médinandi camp that is used as a reference for all surveys done on the Project. Throughout the Project area there are survey reference points that are used as a known reference for the collar surveys. These are included into the round of surveying when surveying new drill collars.

10.11 Downhole Surveys

10.11.1 RC

Depending on ground conditions, and the purpose of the drillhole, RC holes are typically surveyed at 30 to 50 m intervals down hole, using a Reflex down hole surveying rod. If the hole begins to lift / dip the hole can be surveyed at the end of each rod.

10.11.2 Core

Surveys are performed using a Reflex downhole survey (EZ-Track) instrument, with measurements taken at 30 to 50 m intervals down hole. As with the RC drilling, if the hole begins to lift / dip the hole can be surveyed at the end of each rod. Deviation is constantly monitored, and, for example if at the start of a 300 m hole and in the first 50 to 100 m there is significant deviation, the hole may be abandoned and a re-drill undertaken nearby. As an additional check holes are now surveyed every 6 m on exit, after finishing the hole.

10.12 Condemnation, Geotechnical and Hydrogeological Drilling

Geotechnical boreholes were completed using traditional wire line triple tube drilling, typically starting at the ground surface using PQ sized tooling and telescoping down to HQ size core around 230 m depth.

Borehole locations and orientations targeted primarily the interpreted structural features and the final pit slopes, to complement the information previously collected within the mineralized zones.

The following parameters were routinely recorded as part of the geotechnical logging:

- Rock type and geotechnical description.
- Total core recovery (TCR).
- RQD.
- Fracture frequency.
- Weathering / alteration.
- Discontinuity type.
- Discontinuity orientation.
- Discontinuity properties.
- Joint condition rating (Jcon).

Geotechnical logging was carried out for each drill run or for separate geotechnical intervals within a drill run. Bieniawski's rock mass rating (RMR 6) was used to assess the overall quality of the rock to be exposed in the proposed open pit.

Four geotechnical rock units were defined:

- SQR: pink quartzitic schists.
- SST: sandstone.
- ARG: argillite.
- JSP: jasper.

All geotechnical rock unit types were tested using uniaxial (unconfined) compressive strength (UCS) and tri-axial methods.

Minor water bore drilling has been undertaken. Some condemnation drilling in the areas planned for infrastructure has also been completed.

Figure 10.12.1 shows the locations of the geotechnical holes and test pits, and also presents the locations of water bore drilling and other drilling types (e.g. condemnation holes).

Figure 10.12.1 Location Plan, Geotechnical Drilling

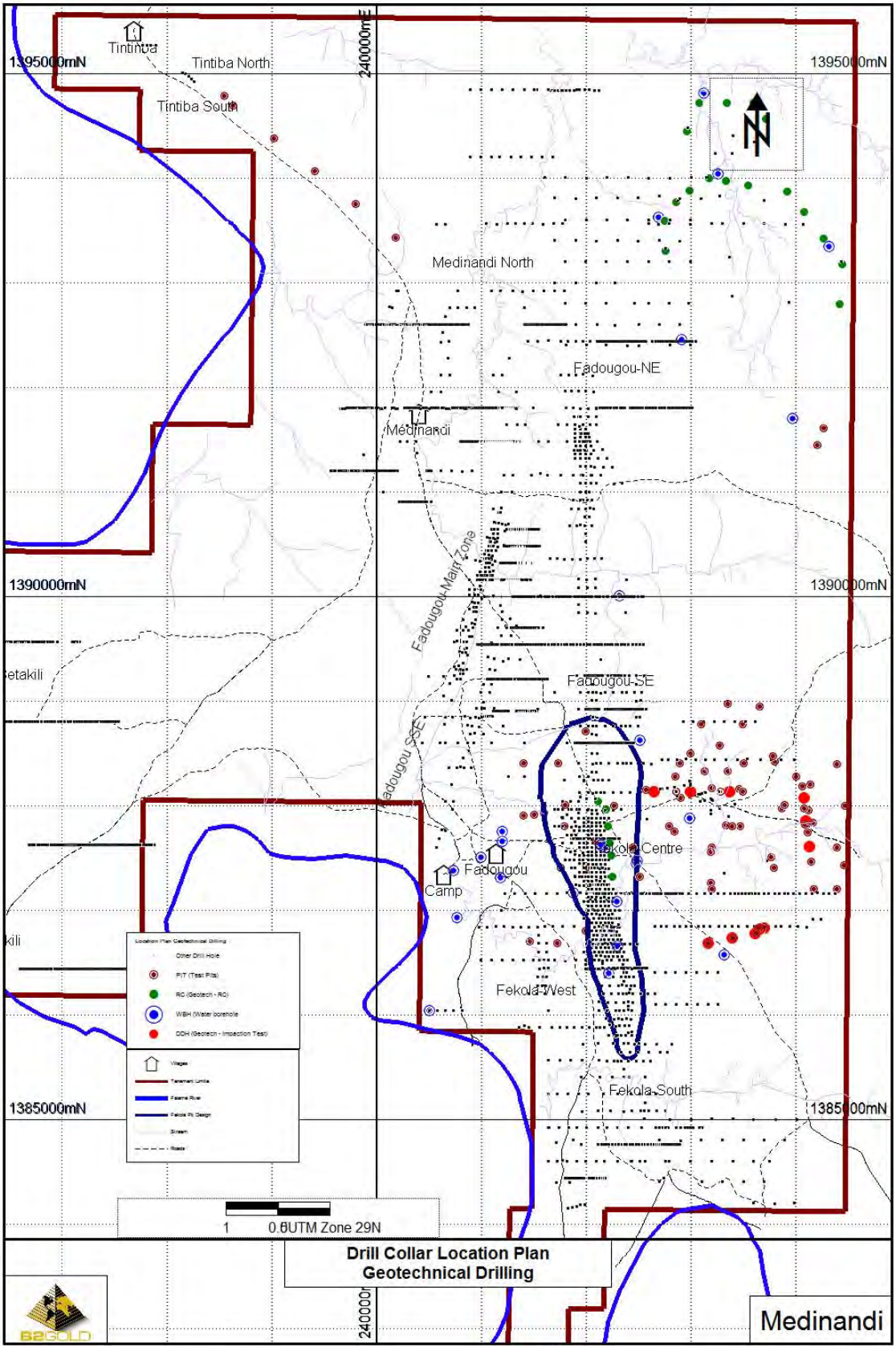


Figure prepared by B2Gold, 2014

10.13 Metallurgical Drilling

Metallurgical drilling is discussed in Section 13.

10.14 Sample Length / True Thickness

Most of the drillholes at Fekola are drilled at -50 to -55° to the east (N90 E) which intersects the main mineralized zone at a high angle. The higher grade mineralisation strikes approximately north–south, is steeply dipping 70 to 80° to the west, and plunges shallowly to the north. In general, true thicknesses are 70 to 80% of the sampled length.

Table 10.14.1 provides a selection of drill intercepts that have been returned from the Fekola deposit. The table illustrates the various grade ranges that have been encountered in the deposit and provides examples of the estimated true thickness of intercepts in relation to the drilled thickness.

Table 10.14.1 Selected Drill Intercept Table

Hole ID	Easting (mE)	Northing (mN)	Total Depth (m)	Dip (deg)	Azimuth (deg)	Intercept From (m)	Intercept To (m)	Drilled Intercept Length (m)	Grade (Au g/t)	True Thickness (m)
FKCR_293	242430	1386278	100	-55	89.5	54	70	16	1.34	11.65
FKD_105	242355	1386480	100	-55	90	27	28	1	2.12	0.75
FKD_105						40	42	2	3.99	1.51
FKD_105						49	57	8	5.43	6.07
FKD_105						68	70	2	1.47	1.53
FKD_105						75	83	8	1.27	6.12
GTD_025	242247	1386862	200	-65	90	10	18	8	1.46	5.1
GTD_025						32	35	3	1.63	1.91
GTD_025						38	40	2	2.32	1.27
GTD_025						42	50	8	4.51	5.11
GTD_025						54	80	26	3.56	16.68
FKD_103	242180	1386960	180	-55	90	29	37	8	2.44	5.84
FKD_103						51	54	3	3.16	2.2
FKD_103						81	145	64	5.12	47.72
FKD_103						146	165	19	1.56	14.28
FKD_055	242060	1387240	309	-55	90	94	98	4	1.23	3.02
FKD_055						106	107	1	2.22	0.76
FKD_055						112	114	2	1.03	1.52
FKD_055						120	122	2	2.41	1.52
FKD_055						144	151	7	2.03	5.35
FKD_055						160	165	5	2.47	3.82
FKD_055						170	277	107	3.84	82.45
FKD_144	241940	1387359	362	-60	83.5	194	197	3	4.46	2.21
FKD_144						216	227	11	1.26	8.14
FKD_144						235	240	5	1.70	3.71

Hole ID	Easting (mE)	Northing (mN)	Total Depth (m)	Dip (deg)	Azimuth (deg)	Intercept From (m)	Intercept To (m)	Drilled Intercept Length (m)	Grade (Au g/t)	True Thickness (m)
FKD_144						245	250	5	1.73	3.72
FKD_144						255	266	11	1.58	8.2
FKD_144						274	311	37	1.48	27.62
FKD_144						333	334	1	2.03	0.76
FKD_048	242020	1387400	320	-55	90	77	78	1	2.22	0.76
FKD_048						91	94	3	1.51	2.28
FKD_048						102	104	2	2.35	1.52
FKD_048						106	107	1	2.10	0.76
FKD_048						115	117	2	1.99	1.53
FKD_048						129	133	4	1.76	3.06
FKD_048						141	142	1	2.62	0.77
FKD_048						152	156	4	2.59	3.08
FKD_048						174	176	2	1.48	1.55
FKD_048						186	227	41	2.68	31.93
FKD_048						231	233	2	2.44	1.56
FKD_048						255	263	8	1.78	6.23
FKD_048						267	269	2	2.03	1.56
FKD_048						282	296	14	2.12	10.95
FKD_121	241780	1387640	600	-55	90	76	77	1	2.97	0.8
FKD_121						253	254	1	1.52	0.81
FKD_121						259	260	1	1.30	0.81
FKD_121						317	318	1	2.14	0.82
FKD_121						332	333	1	1.61	0.82
FKD_121						342	344	2	2.75	1.65
FKD_121						347	351	4	1.07	3.31
FKD_121						392	408	16	3.85	13.19
FKD_121						413	417	4	3.27	3.3

Hole ID	Easting (mE)	Northing (mN)	Total Depth (m)	Dip (deg)	Azimuth (deg)	Intercept From (m)	Intercept To (m)	Drilled Intercept Length (m)	Grade (Au g/t)	True Thickness (m)
FKD_121						424	427	3	2.69	2.47
FKD_121						430	433	3	1.90	2.47
FKD_121						435	440	5	1.92	4.12
FKD_121						464	467	3	3.00	2.48
FKD_121						483	496	13	6.49	10.74
FKD_121						516	517	1	1.96	0.83
FKD_146	241858	1387719	488	-54	86	310	415	105	4.30	82.57
FKD_146						455	456	1	11.60	0.79
FKD_125	241901	1387720	401	-55	83.5	221	226	5	3.80	4.08
FKD_125						237	244	7	3.62	5.73
FKD_125						254	327	73	2.27	60.02
FKCR_276	242359	1386520	66	-55	89.5	24	36	12	11.65	8.79
FKRD_026	241861	1387795	404	-55	89.5	306	381	75	4.66	60.19
FKRD_053	241736.4	1387958.3	555.3	-57	83.5	423.3	426.3	3	1.41	2.23
FKRD_053						482.30	485.3	3	1.42	2.23
FKRD_053						495.30	496.3	1	1.86	0.74
FKRD_053						521.30	535.3	14	1.63	10.57
FKD_148	241793	1388239.2	470.3	-60	-90	121.3	123.3	2	2.21	1.48
FKD_148						192.3	193.3	1	2.51	0.74
FKD_148						295.3	301.3	6	2.06	4.75
FKD_148						401.3	406.3	5	1.28	3.95
FKD_148						421.3	454.3	33	2.90	26.19

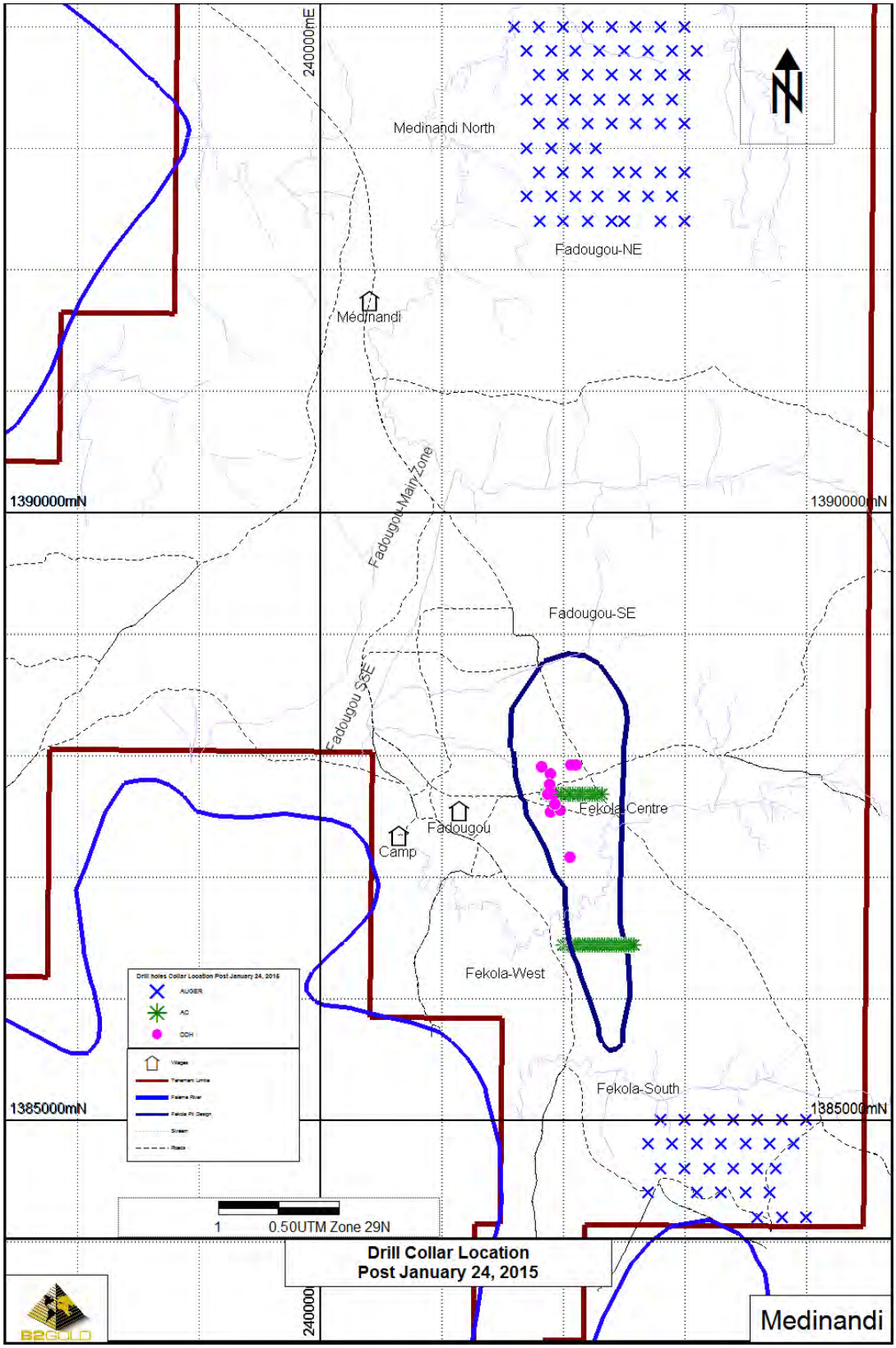
10.15 Drilling Since Completion of Resource Estimates

Drilling has continued after the February 2015 Mineral Resource estimate update and until 13 March 2015. A total of 155 holes (5,263 m) were drilled. Of this total, there are 11 core holes (3,770.2 m), 54 Aircore holes (974 m) and 90 Auger holes (792 m).

Of the total number of holes drilled in this period, 11 holes (3,770.2 m) were infill holes drilled for resource estimation within the Fekola area, 54 aircore holes (974 m) were drilled for assessment of the grade variation within the saprolite across the pit and are close spaced (20 m) and the 90 Auger holes (792 m) were for exploration over prospective areas of the Médinandi licence.

The drilling from this period is shown on Figure 10.15.1. All of the core holes are infill and located within the proposed Fekola pit area. Preliminary results from this drilling support the February 2015 Mineral Resource estimate.

Figure 10.15.1 Drilling Post 24 January 2015



Intercepts are reported using a lower cutoff grade of 1 g/t Au and no upper cut. The true thickness was calculated using a best fit plane of the mineralization oriented at 350º azimuth and -75º dip.

10.16 Comments on Section 10

In the opinion of the QP, the quantity and quality of the lithological, collar, and downhole survey data collected in the exploration and infill drill programs completed, are sufficient to support Mineral Resource estimation and mine planning as follows:

- Core logging meets industry standards for gold exploration.
- Collar surveys have been performed using industry standard instrumentation.
- Downhole surveys were performed using industry standard instrumentation.
- Recovery data from core drill programs are acceptable.
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.
- Drill orientations are shown in the example cross sections included in Section 7, and can be seen to appropriately test the mineralization.

Table of Contents

			Page
11.0	**SAMPLE PREPARATION, ANALYSES AND SECURITY**		**11.1**
	11.1	Sample Preparation and Analysis	11.1
		11.1.1 RC Drilling Samples	11.1
		11.1.2 Diamond Drilling Samples	11.1
		11.1.3 Metallurgical Sampling	11.2
		11.1.4 Analytical and Test Laboratories	11.2
	11.2	Quality Assurance and Quality Control Programmes	11.5
		11.2.2 Standards	11.5
		11.2.3 Blanks	11.5
		11.2.4 Duplicate Samples	11.6
		11.2.5 Magnetic Susceptibility	11.7
		11.2.6 Density	11.7
	11.3	ICP Analysis	11.7
	11.4	Density Analysis	11.7
		11.4.1 Density by Depth	11.8
		11.4.2 Density by Rock Type	11.9
		11.4.3 Density by Mineralized Zones	11.9
	11.5	Portable X-ray Fluorescence Sampling	11.9
	11.6	Sample Security	11.10
	11.7	Sample Storage	11.10
	11.8	Project Drillhole Database	11.10
		11.8.1 Procedures Prior to January 2015	11.10
		11.8.2 Procedures Post-January 2015	11.12
	11.9	Comments on Section 11	11.13

TABLES

Table 11.1.1	Laboratory Summary	11.3
Table 11.1.2	Assay and Grade Population Distribution by Lab	11.3
Table 11.2.1	QAQC Insertion Frequency Summary	11.5
Table 11.4.1	Density by Depth	11.8
Table 11.4.2	Average Density by Lithology	11.9
Table 11.4.3	Average Density by Mineralized Zone	11.9

FIGURES

Figure 11.1.1	Sample Preparation and Analysis Flowsheet (SGS)	11.4
Figure 11.2.1	Sample Preparation Flowsheet	11.6
Figure 11.4.1	Density Variation with Depth	11.8
Figure 11.8.1	Project Data Flowsheet	11.11

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation and Analysis

11.1.1 RC Drilling Samples

RC sampling is carried out by collection of samples through a cyclone into plastic bags.

Samples collected from the rig are brought back to the Fekola camp sample yard. On arrival they are allocated an area depending on sample type and quantity. RC sample bags are placed in down-hole order. The RC samples are collected in batches of approximately 10 sample bags per trip, depending on the speed of the drilling of that particular drillhole.

Sample splitting at the Fekola camp sample yard involves the use of a cone or riffle splitter and a three-tier split. The entire sample must fit in one tray and the number of riffles are maximised (i.e. splitters with a few wide riffles are less effective). The cutter (riffle) width is at least three times the maximum particle size (95% passing).

The target sample weight is 1.5 to 2 kg. If the resultant sample material is under the target weight, the technicians collect whatever available sample there is from the split and that will be dispatched to the laboratory.

Where the cuttings are moist or wet, they are collected into a large calico bag to enable drying before splitting. If a preliminary sample is required, a sub-sample is collected.

Library samples are collected at the riffle splitter and are the resultant sample that is left after the second split. The library sample is kept as a permanent record of the original sample for that interval. The library samples are kept for 12 months after splitting as a reference source if any repeat analytical work is needed.

11.1.2 Diamond Drilling Samples

When a drillhole is selected for sampling a technician goes to the core or sample bags for that drillhole and checks that the depth corresponds to the recorded drillhole depth before allocating sample identification numbers (IDs) for each interval. The sample allocation sheet moves through the sample process, from allocating sample numbers for individual intervals when splitting and cutting, through to sample dispatch.

Core trays returning from the rig are placed in order of drillhole ID and box number on the core tray stands for the geologist to begin logging the core. If the stands are full, the core trays are placed in the designated area in order of drillhole ID and box number until the geologist has finished logging the previous hole and the trays can be moved for further core processing.

A saw cutting line is drawn on the core as a solid red line, and positioned to give symmetry to the sampling. When the core is cut it is split such that there are two half-core pieces that have symmetrical structures or mineralisation.

Core is typically sampled on 1 m average intervals with breaks at lithological contacts and alteration boundaries.

11.1.3 Metallurgical Sampling

Samples used in the Nagrom August 2012 metallurgical test-work were sourced from a ¼-split of HQ-sized core. Samples for the Amdel February 2013 metallurgical test-work had several sources including ½-split of HQ-sized drill core with the exception of composites 1 to 3 which were sourced from sample rejects, and composite 4 which was from the test-work done at Nagrom in 2012. Samples used in the ALS test-work in 2014 and SGS Lakefield test-work in 2015 (run by B2Gold) were sourced from split or dedicated whole HQ-sized core.

11.1.4 Analytical and Test Laboratories

The analytical laboratories used to date are independent commercial laboratories.

From January 2011 to June 2013, the primary laboratory was SGS Kayes, in Mali. Assays from SGS Kayes are the primary support for the January 2013 PEA Mineral Resource estimate, the August 2013 Mineral Resource estimate update, and the current B2Gold Mineral Resource estimate update.

The SGS Kayes facility was closed in mid 2013, and samples were subsequently sent to SGS Bamako in Mali from November 2013. SGS Bamako is the current primary laboratory.

SGS Morila in southern Mali has been used as a secondary laboratory. Primary samples are sent there periodically, and SGS Morila has also occasionally been used for umpire (check) sampling.

Bureau Veritas, Ivory Coast, is currently used as a secondary laboratory and for umpire analyses. Samples selected for multi-element analysis are also shipped through the Ivory Coast laboratory to Bureau Veritas / Acme Vancouver.

Currently there are no commercial minerals laboratories that are fully accredited under ISO17025 in West Africa.

SGS advised that SGS Bamako is currently being assessed by SANAS under ISO17025. The SGS Kayes and SGS Morila laboratories operated a quality system that SGS considered to be in line with ISO17025 requirements.

Bureau Veritas advised that the Ivory Coast laboratory is currently operating to the guidelines of ISO9001 and ISO17025 protocols in accordance with procedures specified within the Bureau Veritas group. The laboratory is currently working on the documentation required to formally have ISO certification in place.

Table 11.1.1 provides a summary of each laboratory's Project involvement by analysis type and year performed, where this is known.

Table 11.1.1 Laboratory Summary

Laboratory	Location	Year	Description
Acme Labs	Abidjan, Ivory Coast	2013 - 2015	Multi-element AR digest; ICP finish; 36 element
Acme Labs	Abidjan, Ivory Coast	2013	Fire assay; 50 g; gravimetric finish
Bureau Veritas	Unknown	2012 - 2014	Fire assay; unknown aliquot; unknown finish
Bureau Veritas	Abidjan, Ivory Coast	2012 - 2015	Fire assay; 50 g; AAS finish
Intertek	Unknown	2011	Fire assay; unknown aliquot; AA finish. Umpires
SGS Bamako	Bamako, Mali	2013 - 2015	Fire assay; 50 g; AAS finish
SGS Kayes	Kayes, Mali	2011 - 2013	Fire assay; 50 g; AAS finish
SGS Lakefield	Ontario, Canada	2014	Fire assay; 50 g; AAS finish
SGS Morila	Morila Mine, Mali	2011 - 2012	Fire assay; 50 g; AAS finish

Table 11.1.2 summarizes the distribution of the assay population and grade contributing to the current Mineral Resource estimate update by Laboratory.

Table 11.1.2 Assay and Grade Population Distribution by Lab

LabCode	Assays	% of Assays	AuPPMSum	% of Au
BureauVeritas	1,795	2	1,163	2
SGS_Bamako	3,144	3	2,149	4
SGS_Kayes	87,601	88	43,825	83
SGS_Lakefield	548	1	1,362	3
SGS_Morila	6,836	7	4,135	8
Sums / Avgs	**99,924**	**100**	**52,634**	**100**

Analyses from SGS_Kayes represent 88% of the total number of analyses contributing to the current Mineral Resource estimate. Analyses from SGS_Kayes represent 83% of the total grade sum of all assays contributing to the current Mineral Resource estimate.

Figure 11.1.1 summarizes the typical sample preparation and analysis flowsheet used by SGS for gold assays for the Project. The main analytical method used for the samples sent for gold assay is fire assay.

Figure 11.1.1 Sample Preparation and Analysis Flowsheet (SGS)

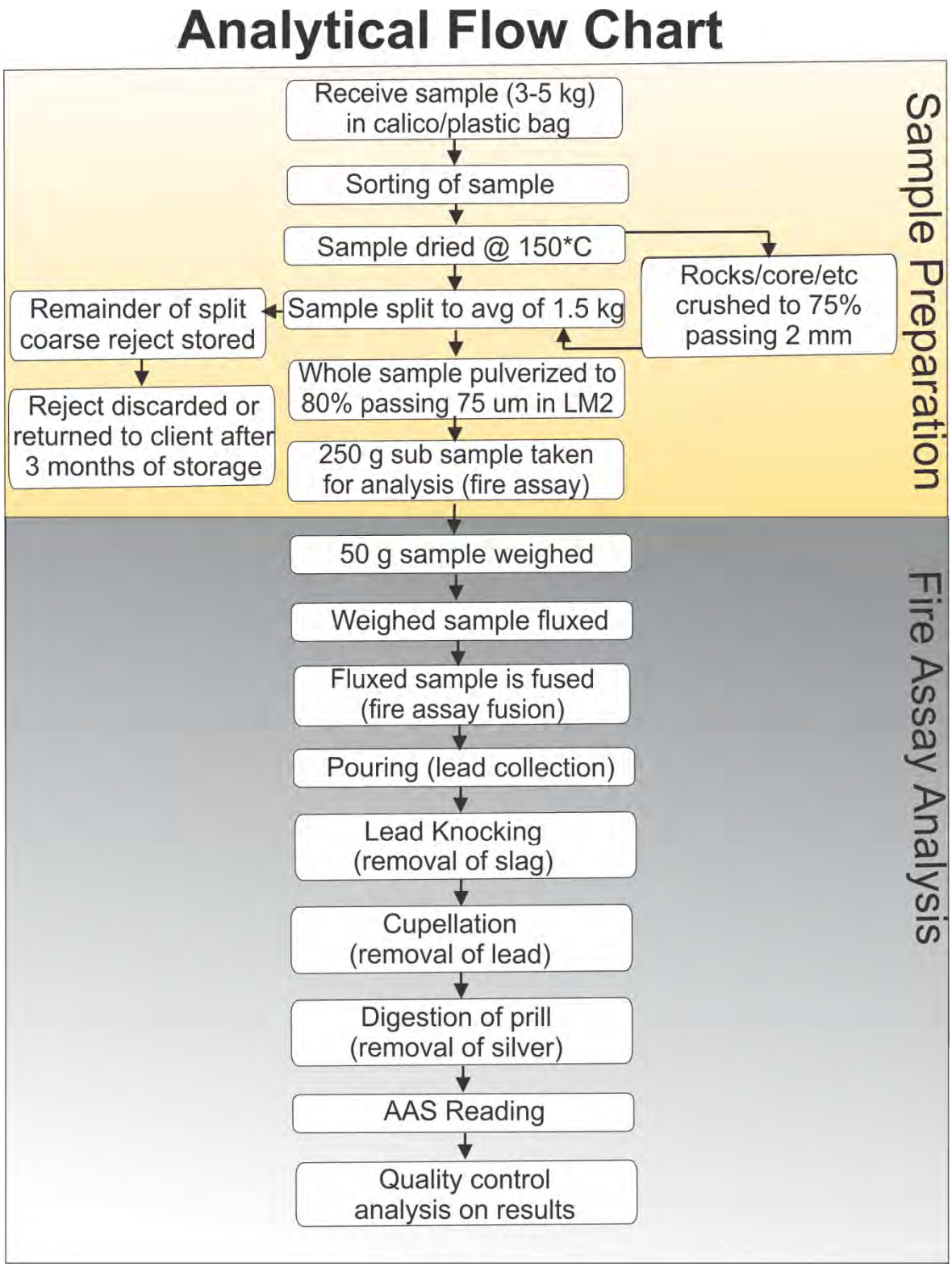


Figure prepared by Papillon, 2013.

11.2 Quality Assurance and Quality Control Programmes

Certified reference materials (CRM or standards), blanks, and duplicates are inserted in the sample sequence at regular intervals to monitor laboratory accuracy and precision and sampling sequencing and precision. Table 11.2.1 summarizes the insertion frequency within the assay population employed for the current Mineral Resource estimate update.

Table 11.2.1 QAQC Insertion Frequency Summary

Insertion Type	Population	Original Samples	% of Orig
Standard	3,355	99,924	3.4
Blank	3,734	99,924	3.7
Fielddup	4,817	99,924	4.8
Prepdup	75	99,924	0.1
Labpulpdup	2,535	99,924	2.5
Labrepeat	7,391	99,924	7.4
Umpire	1,523	99,924	1.5
Sums	**23,430**	**99,924**	**23.4**

Figure 11.2.1 shows a sample submission flowsheet for the Project.

11.2.2 Standards

Five standards are currently in circulation at the project, and cover material from low to high grades. A total of 19 different standards have been employed on the project. Sixteen of these standards, representing ~98% of the population, are Geostats Pty Ltd CRMs. Standards have been inserted at a rate of 1 in 20 samples. The standard insertion frequency was modified in February of 2015 to 1 in 38 samples to suit the SGS Bamako oven batch size.

11.2.3 Blanks

Coarse blanks are submitted with each batch of samples sent to the laboratory. The blank material is either collected by B2Gold (previously Papillon) employees from a location known to be devoid of any mineralization or purchased from a reputable supplier. The non-commercial blank material is collected from barren sandstone material in Kéniéba and is processed on site at Fekola. The blank insertion rate is one in every 20 samples. The blank insertion frequency was modified in February of 2015 to 1 in 38 samples to suit the SGS Bamako oven batch size.

Figure 11.2.1 Sample Preparation Flowsheet

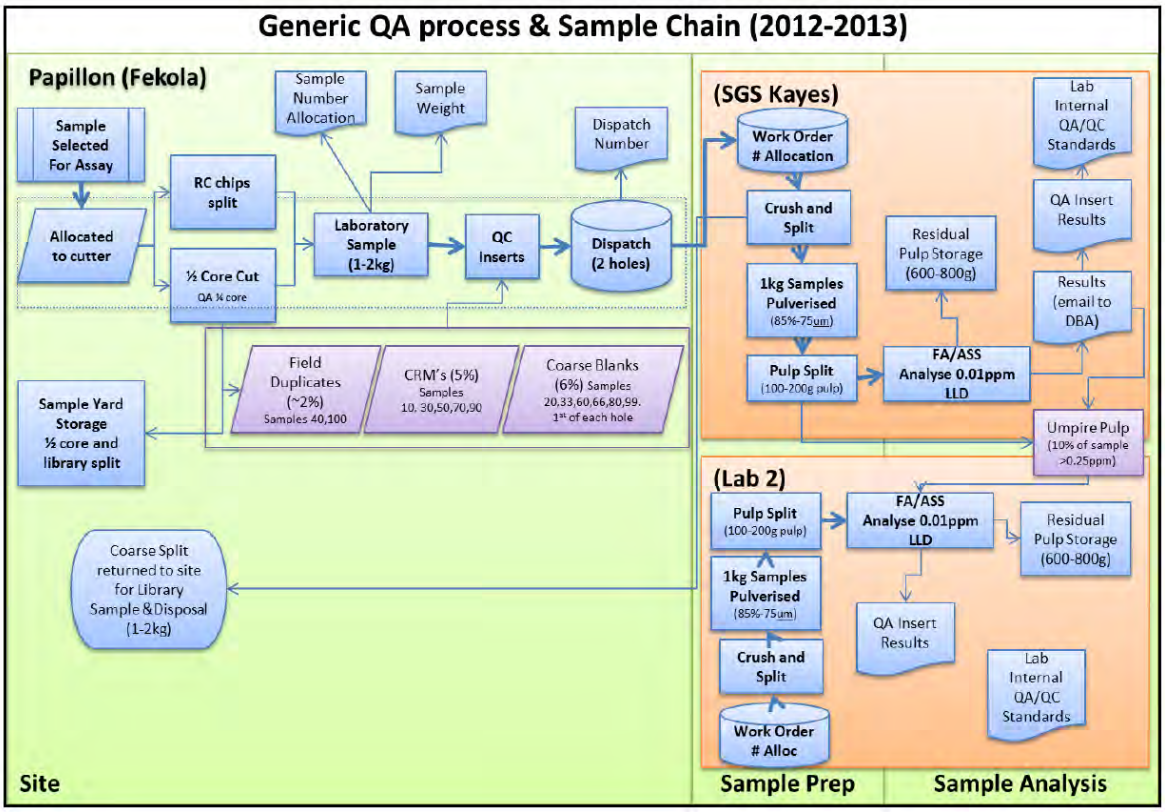


Figure prepared by Papillon, 2013.

11.2.4 Duplicate Samples

Four types of duplicates are run at Fekola to assess the precision of the sampling method and assay analyses; field duplicates (primary sample split, FDUP), pulp duplicates (second split of pulp, SDUP), lab repeats (second fire of primary pulp split, RDUP), and external umpires (secondary lab analysis of pulp reject, UMP). The systematic insertion of preparation duplicates (coarse reject split, PREPDUP) into the core sample stream was implemented in February 2015. Prior to February 2015, preparation duplicates were only taken from the metallurgical test hole FK_MET05.

Field duplicates at Fekola have been inserted at a frequency of one duplicate every 33 samples. The duplicate is collected from the second split at the RC chip splitter or from quartered core. Core field duplicate insertion frequency was halved in February 2015 with the implementation of core preparation duplicates.

Pulp Duplicates and Lab repeats are reported at a minimum frequency of 1 in 76 samples (1 per oven batch). The actual insertion frequency greatly exceeds the minimum (~2 in 76 and ~5 in 76 respectively).

Umpire assays are available for about 1.5% of the assays contributing to the current Mineral Resource estimate update. The relatively low population of umpire assays related to the resource prior to January of 2015 prompted a check umpire submittal in February of 2015. This recent submittal represents about 4% of all assays contributing to the current Mineral Resource estimate update from holes for which there was no existing umpire data and for samples where the original assay grade was greater than 0.25 ppm Au. Umpire assaying procedures have been modified to reflect B2Gold's standard operating procedure of quarterly submittals of ~8% of significant assays.

11.2.5 Magnetic Susceptibility

Magnetic susceptibility is measured with a Terraplus / Georadis KT-10 Magnetic Susceptibility Meter. A section of core with known magnetic susceptibility is used at the start of the drillhole and at the end of the drillhole. Two readings are taken from the core with known values on both sides. A duplicate reading is taken every 1:10 samples. The QAQC data is reviewed periodically. There are 65,264 magnetic susceptibility readings within the database related to holes contributing to the current Mineral Resource estimate update.

11.2.6 Density

A core piece with known weight is used for reference sample readings that are taken at the start of the process and every 25th reading to ensure standard operating procedures are being followed, and weighing scales are being managed correctly. Duplicate density readings are taken every 10th sample in the drillhole.

11.3 ICP Analysis

Multi-element analysis was performed by Bureau Veritas and its predecessor company, ACME, using an aqua regia digest, with an inductively-coupled plasma finish. Approximately 3,537 multi-element analyses, related to the current resource drilling, are available in the database.

11.4 Density Analysis

B2Gold (following Papillon) uses the water immersion method on drill core as a standard procedure for measuring density of rock with low porosity. Measurements are done on air dried whole core samples, typically 10 to 20 cm long. Sample frequency has increased from one sample per 20 to 30 m under Papillon to one sample per 5 m under B2Gold. For near-surface oxidised units, or where core was extremely porous, the volumetric method was occasionally used.

Bulk density is measured by weighing the sample in air and water, and using the formula:

$$M_{air} / (M_{air}-M_{water}) = d$$

Where, M_{air} = mass of the sample in air in grams;

M_{water} = mass of the sample in water in grams.

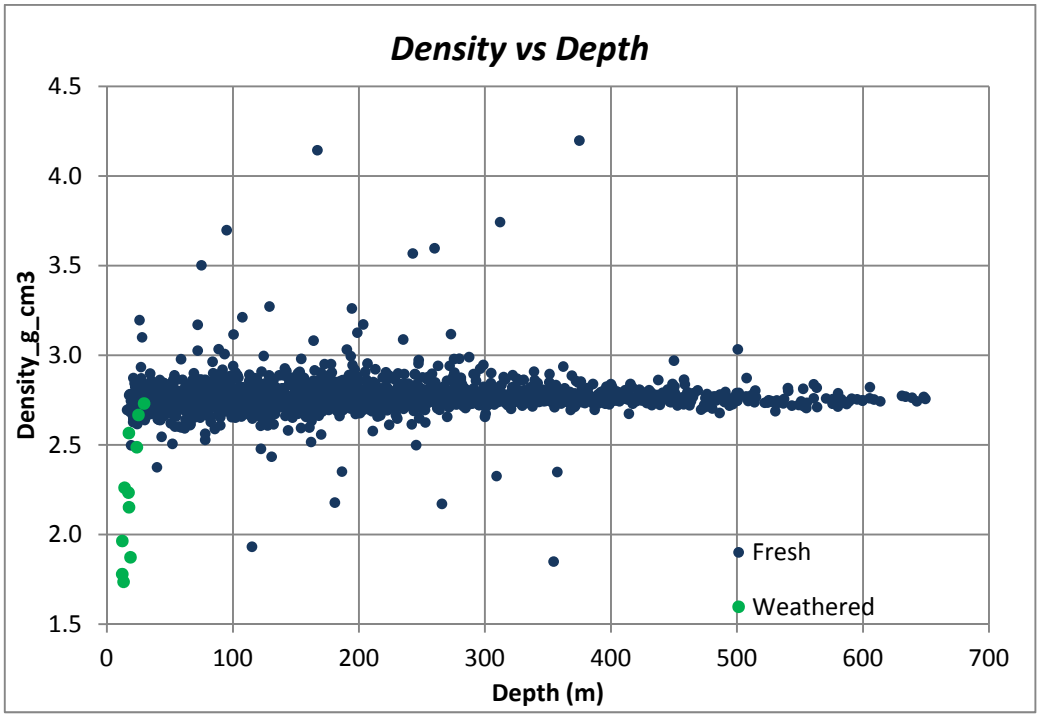
11.4.1 Density by Depth

The current database contains a total of 2,057 original density measurements related to drilling contributing to the current Mineral Resource estimate. Four of these points are characterized as completely weathered or highly weathered and have been excluded from the evaluation population. Density values are summarized by depth from the topographic surface in Table 11.4.1 and on Figure 11.4.1, which support previous observations that there is little variation in density with depth beyond the weathered zone.

Table 11.4.1 Density by Depth

Depth Range (m)	Population	Min. Density g_cm3	Max. Density g_cm3	Mean Density g_cm3
0 – 50	170	1.74	3.20	2.72
50 – 100	304	2.51	3.70	2.76
100 – 150	294	1.93	3.27	2.76
150 – 200	311	2.18	4.14	2.77
200 – 250	271	2.50	3.57	2.77
250 – 300	224	2.17	3.60	2.78
300 – 350	157	2.33	3.74	2.78
>350	322	1.85	4.20	2.77
Sums / Avgs	**2053**	**1.74**	**4.20**	**2.76**

Figure 11.4.1 Density Variation with Depth



11.4.2 Density by Rock Type

Table 11.4.2 presents average density by lithology for dominant lithologies. The population has been filtered to fresh (un-weathered) samples only. The density is quite consistent within the clastic sediments. The MIN unit and the highly dolomitized CCR and SIT tend to have a slightly higher density.

Table 11.4.2 Average Density by Lithology

Lith Group	Lith Code	Lithology	Population	Average Density (g/cm^3)
Argillite	ARG	Argillite	429	2.76
Dyke	MIN	Mafic intrusive	95	2.80
Marble	CCR	Dolomitic mylonite	77	2.82
Mass Flow	MBX	Mass Flow Deposit	195	2.77
Mass Flow	MCG	Meta-conglomerate	106	2.77
Siltite	SIT	Intense dol-ser-py alteration	111	2.78
Turbidite	BSM	Siltstone/Mudstone	1,006	2.76
All			**2,106**	**2.77**

11.4.3 Density by Mineralized Zones

Average density by mineralized zone (see Section 14.4) shows a slight increase in density with the higher grade zones, as shown on Table 11.4.3.

Table 11.4.3 Average Density by Mineralized Zone

Mineralized Zone	Total Samples	% of Total	Avg Au	Avg Density
Outside GZ101	919	44.85	0.03	2.75
GZ101 (0.1 to 0.5g/t)	527	25.72	0.23	2.76
GZ105 (0.5 to 1.0g/t)	264	12.88	0.76	2.77
GZ110 (1.0 to 2.5g/t)	187	9.13	1.63	2.79
GZ125 (>2.5g/t)	119	5.81	6.80	2.83
(blank)	33	1.61	0.06	2.73
Grand Total	**2049**	**100.00**	**0.72**	**2.76**

11.5 Portable X-ray Fluorescence Sampling

An Olympus Innov-X Delta portable X-ray fluorescence (PXRF) instrument and workstation is used to determine relative concentrations of various elements within sample pulp rejects. Historically, samples were scanned through the kraft paper bag in which the pulp was returned from the assay laboratory. It was subsequently determined that this process results in significant signal attenuation. Results from this earlier process have been rejected in the database.

The pulps are being re-scanned through a thin plastic film sachet. This process is ongoing. Of the ~28,000 valid raw XRF results in the database, ~12,500 are from samples contributing to the current resource update.

The XRF scanning sequence incorporates blanks, commercially available standards, and duplicates.

The process of preparing matrix matched standards from deposit material is underway. These samples will be used to determine calibration / correction curves that may be applied to the raw data to more accurately reflect absolute concentrations.

The captured data is stored within the drillhole database in its raw/uncorrected format.

11.6 Sample Security

Sample security measures include moving RC samples and core from the drill site to the Fekola camp yard at the end of each drill shift and tracking of sample shipments using industry-standard procedures. B2Gold is of the opinion that core storage is secure because the camp is remote, access is strictly controlled and a B2Gold (previously Papillon) representative has always been in the camp.

11.7 Sample Storage

Pulp rejects are returned from the assay lab to site. Pulps are stored in bar-coded kraft paper pulp bags within cardboard pulp trays/boxes within shipping containers allocated for that purpose.

Drill core is stacked in open-sided sheds.

RC drill chips are stored in lidded, plastic core trays, most of which are kept in a shipping container / shed. The first tier RC split is referred to as a library sample, and is stored in the Fekola camp yard for 12 months prior to discard.

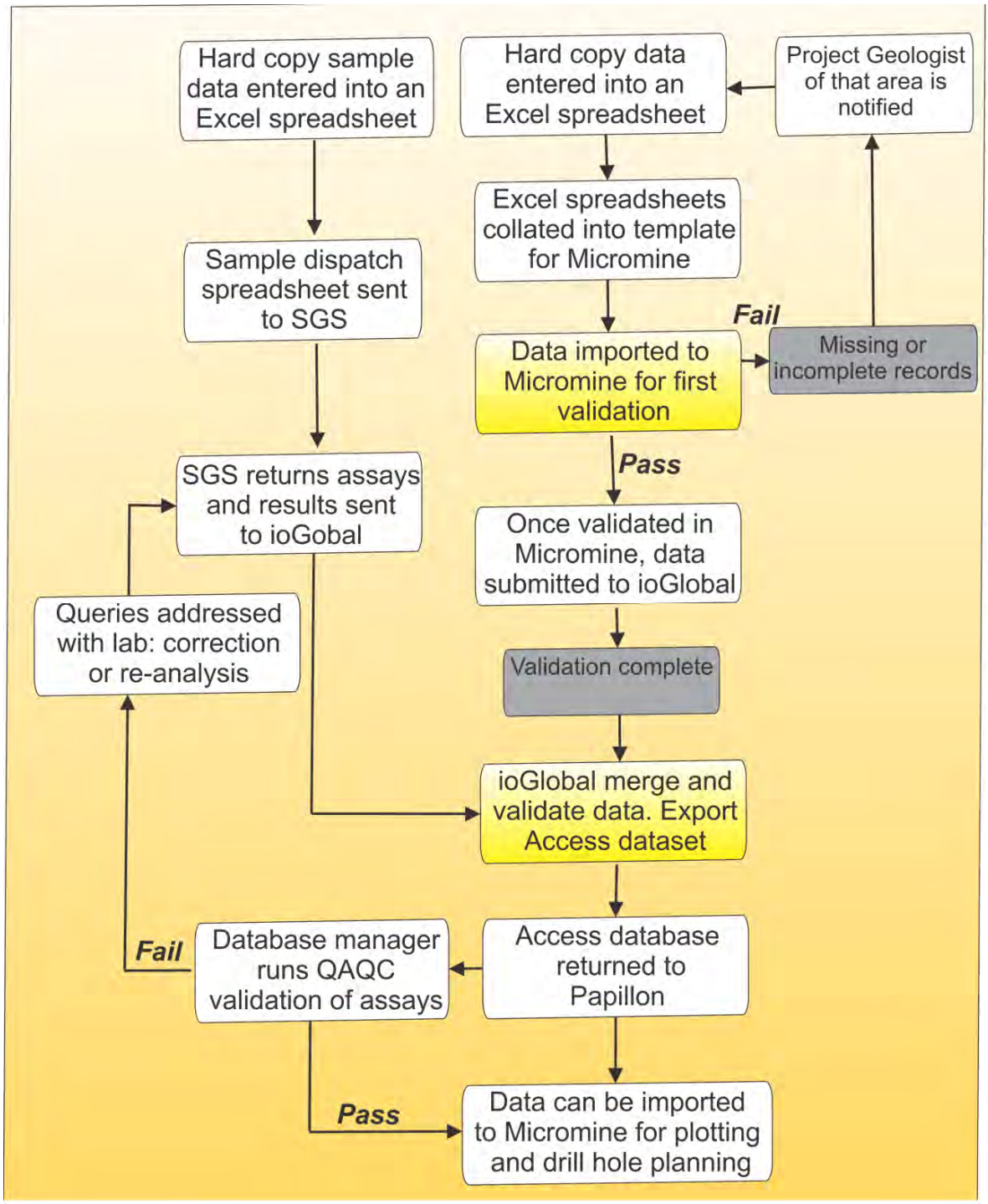
11.8 Project Drillhole Database

11.8.1 Procedures Prior to January 2015

Prior to January of 2015 the project drillhole database was managed remotely by IoGlobal in Perth, WA. All of the drillhole data contributing to the current resource update was originally managed in the IoGlobal system which was implemented in August 2010. FKRC_008, the Fekola discovery hole, was drilled in December 2010, so all data from holes drilled by Papillon in the resource area was managed in the ioGlobal system.

The data flow outlined below is specific to the IoGlobal procedures in place prior to January of 2015. Figure 11.8.1 is a Project data flowsheet.

Figure 11.8.1 Project Data Flowsheet



All field data at Fekola is initially captured on paper. Data entry personnel enter the data into a series of Excel templates with extensive pick-lists and validation rules. The drill geologist checks the digital file against the paper original and the original hole proposal. The data is imported into Micromine for a first-stage validation. This stage checks that drillhole collars are plotting in the correct locations and that loaded drillhole data is complete. The original paper capture forms are filed on site.

The data was uploaded to the Papillon SharePoint. IoGlobal had access to the folder location on the Papillon SharePoint and were able to download the prepared data files. IoGlobal imported the data into the Papillon database; during this process the data was validated. If there were any issues with the data, ioGlobal highlighted the issues to the database manager for correction. Exports from the ioGlobal database were in Access format. The updated database was loaded on the ioHub; this is the ioGlobal cloud-based data transfer location. The database manager subsequently downloaded the updated database from the ioHub.

Assay data was received from SGS and sent directly to ioGlobal by the database manager for merging into the Access database. Once merged the assay data was sent back to Papillon as part of the Access database. At this stage a QAQC report was generated to validate the assays from SGS, and if there were errors the database manager queried this with the laboratory (see also Section 12).

Once the database manager had addressed all errors and problems highlighted by ioGlobal and the QAQC report, the database was handed over to the managing geologist for import into Micromine software for further analysis.

11.8.2 Procedures Post-January 2015

The transition to onsite / in-house database management was completed in January of 2015 by B2Gold employees. The database is currently managed onsite in the B2Gold standard database format. The data flow has not changed substantially except for the elimination of the remote hosting.

As with the IoGlobal system, all field data at Fekola is initially captured on paper. Data entry personnel enter the data into a series of Excel templates with extensive pick-lists and validation rules. The drill geologist checks the digital file against the paper original. The data is imported into Micromine for a first-stage validation. This stage checks that drillhole collars are plotting in the correct locations and that loaded drillhole data is complete. This process is overseen by the database manager. The original paper capture forms are filed by drillhole.

The onsite database manager, a geologist, imports the data into a purpose built MS-Access database, and checks the data by inspection and with validation queries built into the database.

Assay data is imported as text, retaining the original lab codes, and is translated to numeric values within the database. Assay results are not associated with samples until the results have been QAQC vetted. Assay results for blanks and standards are compared with expected results via queries in the database. Subsequent to QAQC validation assays are assigned a Passing (1) or Failing (3) priority. Failed assays are excluded from database exports.

Export subsets, in the same MS-Access format as the IoGlobal exports, are generated by macros within the database. These files are created and published to an online file transfer portal subsequent to any significant change within the database.

The database includes QAQC reporting utilities to facilitate tracking standard and blank performance, duplicate precision, and analytical bias. QAQC data is reviewed on a continuous

basis as data is imported into the database. Comprehensive QAQC reports are generated and reviewed by senior staff on a monthly basis.

The entire database is backed up to an online file transfer portal twice weekly. These backups are downloaded and stored on the B2Gold file server in Vancouver on a regular basis.

Digital photos are stored on the site server and identified by drilling method and drillhole ID. Each drillhole is photographed wet and dry, and the picture is named with drillhole ID and interval. Digital images are backed up to a separate location to the primary database.

11.9 Comments on Section 11

In the opinion of the QP's:

- Sample collection, preparation, analysis and security for RC and core drill programs are in line with industry-standard methods for gold deposits.
- Drill programs included insertion of blank, duplicate, and standard reference material samples.
- QAQC methods are practiced during magnetic susceptibility and specific gravity measurement programs, which are industry leading practices.
- QAQC program results do not indicate any problems with the analytical programs (refer to discussion in Section 12).
- Data is subject to validation, which includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards (refer to discussion in Section 12).
- All core and RC chips have been catalogued and stored in designated areas.

The QP is of the opinion that the quality of the gold analytical data is sufficiently reliable to support Mineral Resource estimation without limitations on Mineral Resource confidence categories.

Table of Contents

Page

12.0 DATA VERIFICATION **12.1**
12.1 QAQC Data Reviews 12.1
12.1.1 Papillon / B2Gold Data Checks 12.1
12.1.2 ioGlobal Data Checks 12.4
12.1.3 Results 12.4
12.2 QAQC Analysis 12.4
12.2.1 MPR Reviews, Mineral Resource Estimate Update Data Support 12.4
12.2.2 B2Gold QAQC Review 12.6
12.2.3 Comments on Section 12 12.18

TABLES
Table 12.2.1 Standard Analytical Bias 12.11
Table 12.2.2 Field Duplicate Grade Sums 12.12
Table 12.2.3 Field Duplicate Grade Sums by Grade Range Bin 12.13
Table 12.2.4 Umpire Population by Lab 12.15
Table 12.2.5 Umpire Population Grade Sums by Lab 12.16
Table 12.2.6 SGS_Kayes vs Bureau Veritas Umpire Grade Sums 12.18
Table 12.2.7 Bureau Veritas vs SGS_Kayes Bias 12.18

FIGURES
Figure 12.2.1 Fekola Pulp Duplicate Scatterplot 12.7
Figure 12.2.2 Fekola Pulp Dups Thompson-Howarth Plot 12.8
Figure 12.2.3 Fekola Lab Duplicate Scatterplot 12.9
Figure 12.2.4 Fekola Lab Dups Thompson-Howarth Plot 12.9
Figure 12.2.5 Fekola Blank Performance 12.10
Figure 12.2.6 Fekola Field Duplicate Scatterplot 12.13
Figure 12.2.7 Fekola Field Dups QQ Plot 12.14
Figure 12.2.8 Fekola Field Dups Thompson-Howarth Plot 12.15
Figure 12.2.9 Fekola Umpires: Kayes vs BV, BV vs Bamako 12.16
Figure 12.2.10 Fekola Umpires: BV vs Morila 12.17
Figure 12.2.11 Fekola Umpires: BV vs Kayes 12.17

12.0 DATA VERIFICATION

12.1 QAQC Data Reviews

12.1.1 Papillon / B2Gold Data Checks

Data Entry

All field data at Fekola is initially captured on paper. Data entry personnel enter the data into a series of Excel templates with extensive pick-lists and data validation rules. The drill geologist checks the digital file against the paper original and the original drillhole proposal.

The data is then imported into Micromine for a first-stage validation. This stage checks that the drillhole collar is plotting in the expected location with the correct geometry and that the drillhole data is complete.

Prior to the January 2015 database management system change, subsequent to initial validation and correction, the template was placed in the 'prepared' templates folder. The database manager would check the prepared templates and load them to the Papillon SharePoint folder for ioGlobal to download. Once the template was loaded to SharePoint the template was moved from the prepared folder to the 'sent templates' folder.

Subsequent to the January 2015 database management system change by B2Gold, the data is imported directly into the onsite drillhole database for additional validation checks after the initial Micromine validation.

QAQC

QAQC data is reviewed on a continuous basis as data is imported into the database. Comprehensive QAQC reports are generated and reviewed by senior staff on a monthly basis. Actions prompted by the review are implemented, and again reviewed at the end of the following month.

Papillon employed the following tolerances as the basis for evaluation of the QAQC data:

- The laboratory Certified Reference Material (CRM) or standard checks should have a normal distribution since the analyses are done and compared on the same standard. A 5% tolerance is given either side of the expected mean of the CRM. This provides a visual guide for where the data should be plotting. Samples falling outside the ±5% are reviewed individually. In theory, approximately 5% of results fall outside the expected limits.

- Approximately 62% of the values should lie within one standard deviation of the mean, with approximately 95% of values within two standard deviations or 5% and 99% should fall within three standard deviations of the mean.

Any outliers are investigated. Instances that are significantly outside the tolerance lines are investigated in the following order of assumption:

- Was the sample number incorrectly assigned?
- Sample handling errors – laboratory mix up of samples or sample numbers.
- Grades of neighbouring samples and CRMs reviewed to determine if values fall within the expected CRM range.
- The laboratory is requested to review its internal QAQC.
- Consideration is given to re-analysing surrounding samples.
- If there are three or more CRMs that are considered outliers, the batch is considered for re-analysing.

QAQC data reviews include:

- Standard plots: assay standard values are compared against expected values for each standard; assay values are compared against the expected null value for blanks.
- Correlation plots: original assay data are compared to repeat assay data.
- Scatter plots: compare field duplicates and laboratory duplicates against the original sample.
- Target plots: used to measure accuracy and precision of CRMs.
- Check laboratory comparison plots: umpire samples are plotted against assay results from the primary laboratory.

B2Gold's QAQC Procedures

With the transition to onsite / in-house database management B2Gold QAQC validation rules have been implemented:

- A standard analysis more than three standard deviations (SD) from the expected value constitutes a failure (3SDHIGH or 3SDLOW).
- A standard analysis between two and three standard deviations from the expected value generates a warning (WARNHIGH or WARNLOW).
- Two sequential standards more than 2SD from the expected value on the same side of the expected value constitute a bias failure. (BIASHIGH, BIASLOW).

- The published standard deviation of artificial standards is often very small (as a percentage of the expected value). B2Gold employs limits based on the published standard deviation or 3.33% of the expected value whichever is greater. This practice is based on the verbal advice of Dr. Barry W. Smee, Ph.D., P.Geo. (Smee and Associates Consulting Ltd). The published standard deviation has been employed for the majority (17 of 19) of the standards employed by Papillon. AUSK_1 (35 instances) and Si54 (16 instances) use a calculated SD of 3.33%.

- Blank analyses generate warnings at five times the method detection limit and failures when they exceed 10 times the method detection limit.

The treatment of failures is similar to that previously employed by Papillon. The failure is evaluated in context of the surrounding samples and standards to determine if the failure is possibly the result of a standard or blank misidentification or if the failure is possibly the result of a laboratory mix up of samples or sample numbers. Sample weights, assays, and Z-scores (deviation of the analysis from the expected value in terms of number of standard deviations) are all used to aid this type of evaluation.

If it is not possible to determine, with reasonable certainty, that the failure is the result of a standard substitution error or a sample mix-up, either in the submittal or at the lab, re-analyses of all or part of the batch may be requested.

In cases where the failure is within a sequence of low grade/insignificant results the project manager may elect to accept the original certificate, regardless of the failure, on the basis that it will not significantly affect any resource data.

QAQC data is reviewed on a continuous basis as data is imported into the database. Comprehensive QAQC reports are generated by the site database manager and reviewed by senior staff on a monthly basis.

Monthly QAQC reports include:

- A summary of the samples shipped and analyses received during the period with a statement of turnaround time.

- Standard and Blank performance summaries by month, year, and standard type.

- Standard analysis bias summaries by grade bin.

- Individual standard performance tracking plots. Standard assays are tracked over time for each standard. Separate plots are generated for initial values, including failures, and final accepted values. These plots permit evaluation of precision, accuracy, and bias trends for individual standards.

- Blank performance tracking plots: blank assays are tracked over time. Separate plots are generated for initial values, including failures, and final accepted values. These plots permit identification and evaluation of anomalous trends in blank assay performance.

- Duplicate scatterplots: scatterplots are presented for Field duplicates (primary sample split, FDUP), Preparation duplicates (coarse reject split, PREPDUP), Pulp duplicates (second split of pulp, SDUP), and Lab repeats (second fire of primary pulp split, RDUP).

- Thompson-Howarth plots are presented for all duplicate types. These plots allow evaluation of sampling / analytical precision.

- Umpire samples are submitted and evaluated on a quarterly basis.

12.1.2 ioGlobal Data Checks

ioGlobal (during Papillon's tenure) produced a monthly QAQC report that was generated and sent to the database manager. A list of missing sample and assay data was also prepared at the end of every month.

Sample information sheets were uploaded to ioGlobal to download. Within the sample sheet, samples that had not been sent for assay were indicated with -3333 in the Au result column; this allowed ioGlobal to distinguish which samples were not erroneous but were actually awaiting processing, or had been intentionally missed.

12.1.3 Results

Examination of the QAQC sample data indicates satisfactory performance of field sampling protocols and assay laboratories providing acceptable levels of precision and accuracy.

12.2 QAQC Analysis

12.2.1 MPR Reviews, Mineral Resource Estimate Update Data Support

MPR completed a QAQC review as part of their February 2013 and August 2013 resource estimation work; the following summarizes the review completed at that time.

First Duplicates

The drillhole assay database contains first duplicate analyses for 8,360 samples, equating to approximately one duplicate assay for every 15 original assays. Scatter plot evaluation of the original gold assays with assays of the pulp duplicates data indicated the means of the original and duplicate assay pairs are the same at 1.16 g/t Au, suggesting that sample extraction and analytical errors is low.

Blanks

Insertion of blank material is part of the routine QAQC protocol and the 5,052 available blank assays equate to a frequency of 1:25 blank to unknown samples. The assaying of blank material has returned near 0.00 g/t Au for all but seven samples. These higher than expected grades range from 0.17 to 3.68 g/t Au and are suspected to be mislabelled standards.

Standards (CRMs)

Nineteen certified reference materials, sourced mostly from Geostats Pty. Ltd. (Geostats), have been routinely inserted within the assaying batches at Fekola.

There is a generally good agreement between average assayed grades and recommended grades for all standards. All standards have assayed average grades within 5% of the recommended grade with the exception of two lower grade standards (G310-3 and G303-8) and these are within 0.01 ppm (equivalent to the assay quantisation) of their expected grades. More importantly there is no trend in the small variances towards high or low bias.

Core and RC Duplicates

There are 2,990 pairs of diamond core duplicates. The global means of the two data are comparable and show no grade bias towards either pair. A QQ plot indicated no conditional bias. The result is considered acceptable by MPR given the type of comparison.

There are 2,651 pairs of RC field duplicates. The means of the two data are identical at 0.36 g/t Au. The QQ plot shows that there is no bias towards either sample pair. The result is considered excellent by MPR.

Twin Hole Drilling

Papillon completed a program of diamond core holes twinning previously drilled RC holes. This was initiated in 2011 to 2012 after concerns were raised on the possible poor representivity of wet RC sampling.

Visual inspection of all sections shows that there is good correlation between the intersection of gold mineralisation seen in the RC drillholes and their neighbouring diamond core twins. Another encouraging observation is that in the broader intersections where there are internal zones of lower-grade mineralisation the contacts to these zones (and grade tenor) are reflected similarly in the two sampling methods.

To confirm the visual comparison MPR initially calculated gold intersections (down-hole length and grade) for each drillhole twin. The intersections were based on a nominal 0.2 g/t Au cut-off and sample grades were not cut prior to calculating the average grades.

In most cases the mineralised widths intersected in the twins are similar. The average grades of the mineralisation do show some variance, but this to be expected, particularly in higher-grade gold mineralisation, and is also partly due to the hole paths deviating in some instances.

The total accumulated length of mineralisation selected from the RC drilling is 792 m, averaging 2.56 g/t Au. The equivalent selected diamond core hole intersections sum to 796.5 m with an average grade of 2.52 g/t Au and compares very well with the cumulative RC average grade.

As an additional check to complement the visual and intercept comparisons of the RC and diamond core twins, a file of nearest-neighbours was generated and the resultant data pairs analysed. Within a search ellipse of 10 m (east) x 10 m (north) x 5 m (elevation), centred on an RC sample,

pairs of RC and diamond core samples were selected. The complete dataset contains 914 pairs with pairs having an average separation distance of 7.5 m.

The pairs of RC and core sample grades were plotted, based on an upper grade cutoff of 12 g/t Au, and excluding 15 pairs. The scatter on the resultant figure is broad and the data have a poor correlation coefficient (linear = 0.43). These are expected given the extreme short scale discontinuity generally seen in gold. Importantly the mean grade of both data pairs is very similar; 0.82 versus 0.83 g/t Au, RC and core, respectively. The QQ plot shows no significant conditional bias towards either sample type, although there is a suggestion that at lower grades the RC samples tend to be higher in grade than their core sample counterpart. The reverse is true for grade above 4.0 g/t Au. In both cases, the bias is less than 10% and not considered to be a significant area of concern.

Review Results

MPR was the resource geologist for prior work, including the PEA filed by B2Gold in August 2014. MPR noted, following the reviews, that:

- The QAQC data available that support the drill information at Fekola do provide assurance to Papillon that the assays informing the resource model and do not display sampling or assaying biases.
- The assaying of routine laboratory duplicates, blanks and the results of field duplicates (RC and diamond) are all conforming to or exceeding accepted industry standards. The results of assaying the certified standards within batches of the originals and QAQC samples suggest there is no bias in the assaying.
- The most conclusive data are the results from the diamond core twinning of the RC intercepts.

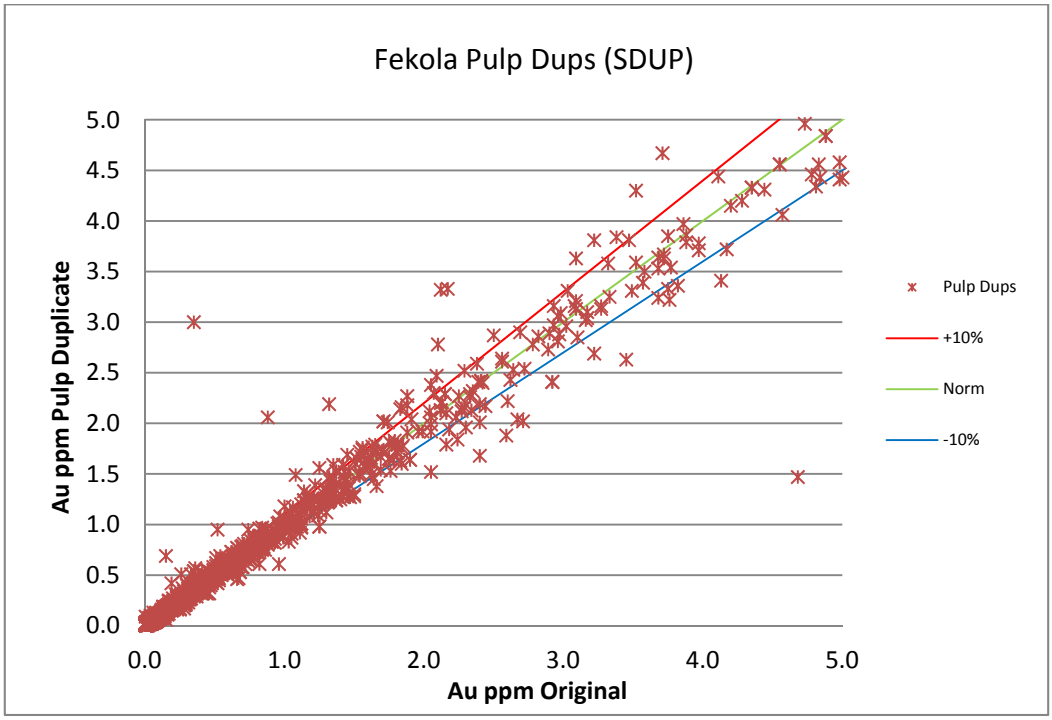
12.2.2 B2Gold QAQC Review

The cutoff date for the drillhole database used for the mineral resource estimate was 24 January 2015. This was the same dataset used in the QAQC analysis described in the following sections.

Pulp Duplicates (SDUP)

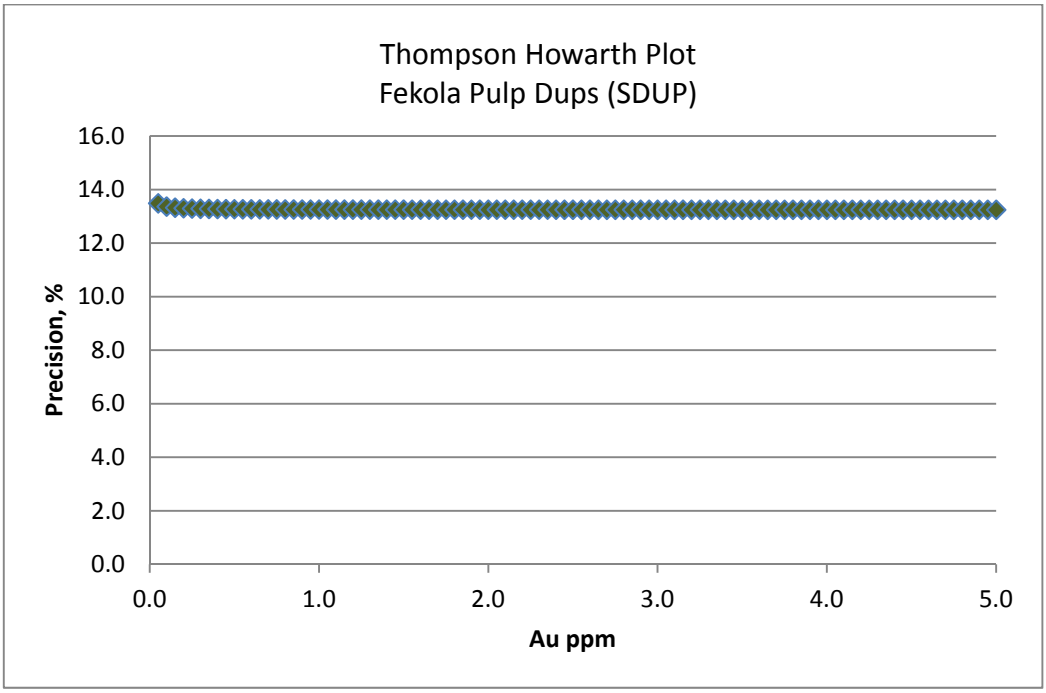
There are 2,535 pulp duplicate analyses (second split of pulp), which is about 2.5% (or ~1 in 40 samples) of the original sample population contributing to the current resource update. Scatterplot (Figure 12.2.1) evaluation of the pulp duplicate pairs shows a good correlation between the original assays and the duplicates with very few outliers (eight pairs with RSD >100%).

Figure 12.2.1 Fekola Pulp Duplicate Scatterplot



A Thompson-Howarth plot of the data (Figure 12.2.2) is very flat with precision at 13.3% above 0.15 ppm Au. The total grade sum of the original assays (2,603.2 ppm) compares very well with the total grade sum of the pulp duplicate assays (2,601.4 ppm).

Figure 12.2.2 Fekola Pulp Dups Thompson-Howarth Plot



Lab Duplicates (RDUP)

There are 7,391 lab repeat analyses (second fire of primary pulp split), which is about 7.4% (~1 in 14 samples) of the original sample population contributing to the current resource update. Scatterplot (Figure 12.2.3) evaluation of the lab repeat pairs shows a good correlation between the original assays and the duplicates with very few outliers.

A Thompson-Howarth plot of the data (Figure 12.2.4) shows a precision of 16.6% at 0.50 ppm Au. The total grade sum of the original assays (10,313 ppm) compares very well with the total grade sum of the pulp duplicate assays (10,300 ppm).

Figure 12.2.3 Fekola Lab Duplicate Scatterplot

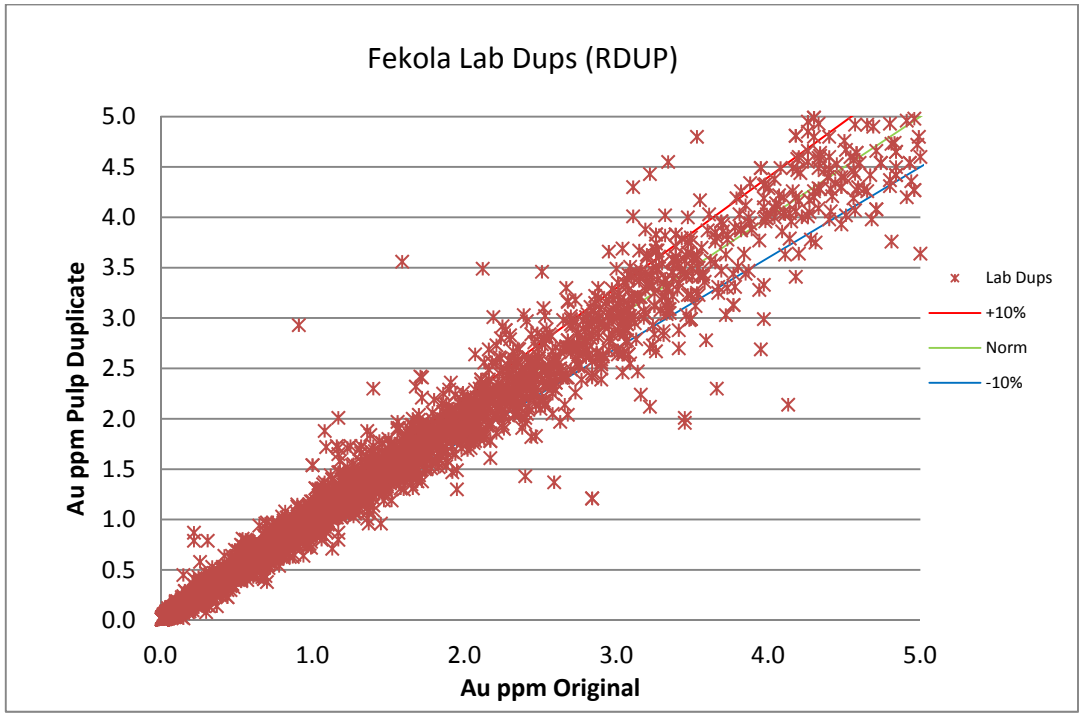
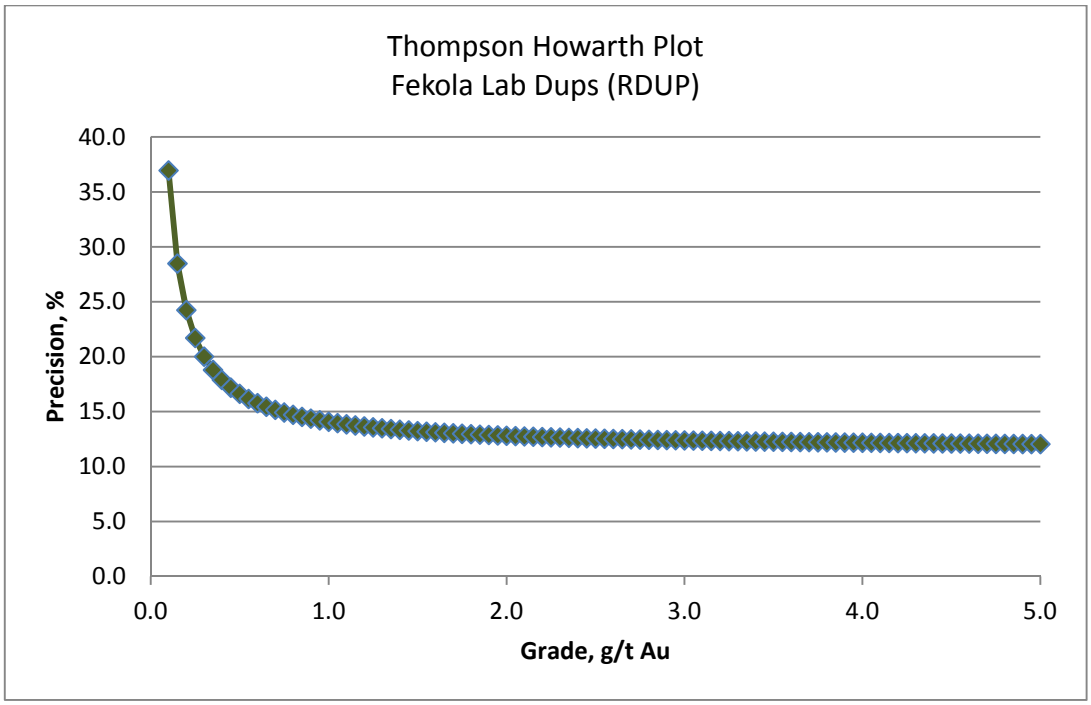


Figure 12.2.4 Fekola Lab Dups Thompson-Howarth Plot



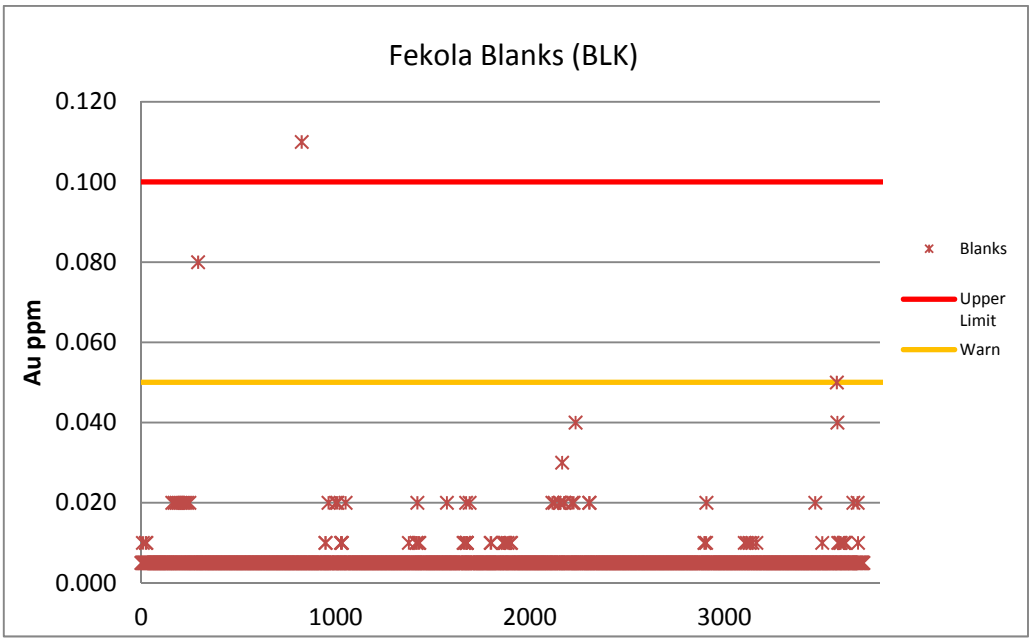
The positive correlation and high precision of the pulp duplicate and lab duplicate pairs suggests that analytical and sampling errors at the pulp level are very low.

Blanks

There are 3,734 blank (coarse preparation blank) analyses, which is about 3.7% (~1 in 26 samples) of the original sample population contributing to the current resource update. The assaying of blank material has returned near 0.00 g/t Au for all but two samples (note that four insertion substitution errors involving standards originally identified as blanks were corrected in January of 2015.). Blank SORES_E0005080 in SGS_Kayes certificate KY098619 generates a warning message ("WARNHIGH") with a value of 0.08 ppm Au. Blank SORES_D012460 in SGS_Kayes certificate KY098796 generates a failure message ("FAILHIGH") with a value of 0.11 ppm Au. The cause of these two anomalies has not been identified. It is possible that these represent sample mix-ups. The low grades involved preclude any significant effect on the resource update.

The very low frequency of blank failures suggests that the laboratories are working cleanly with few sample mix-ups.

Figure 12.2.5 Fekola Blank Performance



Standards (CRMs)

There are 3,355 standard (Certified Reference Material, STD) analyses, which is about 3.4% (~1 in 30 samples) of the original assay population contributing to the current resource update. The majority, 2,933, of these standard analyses are related to SGS_Kayes analyses.

A total of 19 different standards have been employed on the project. Sixteen of these standards, representing ~98% of the population, are Geostats Pty Ltd CRMs. It should be noted that 16 standard misidentification errors were corrected in January of 2015.

There are six standard failures, by B2Gold validation criteria, related to analyses that remain in the population of assays contributing to the current resource update. Two of these failures are related to a very low grade, 0.07 ppm, standard; G310-3. A single 3SDHIGH failure on a 0.51 ppm standard is associated with insignificant results. Three very consistent analyses of a 2.88 ppm standard (2.50, 2.51, and 2.53 ppm) resulting in 3SDLOW failures may represent a standard misidentification.

In all cases the net effect of accepting the analyses with the included failures would be to understate the grade or have little effect on any significant assays.

Table 12.2.1 presents average Z-Scores for various standard grade range bins for each of the laboratories contributing to the assay database. Note that the failed population of six standards has been removed from this population.

The Z-score is the number of standard deviations the analytical result deviates from the expected value. The weighted average standard deviation of the standards employed by Papillon is 8%. An average Z-Score of -0.1 implies a low bias of ~0.8%.

The figures in Table 12.2.1 suggest that there is little significant bias in the standard analytical results. The limited populations of SGS_Lakefield analyses show a moderate overall positive bias. The limited population of SGS_Morila analyses show a moderate negative bias. Bureau Veritas and SGS_Bamako bias trends are effectively neutral. The SGS_Kayes standard analyses, associated with assays representing 88% of the total gold assay population, show a slight negative bias below 2 g/t, a slight positive bias between 2 g/t and 4 g/t, and a moderate positive bias in the small population of results exceeding 4 g/t.

Table 12.2.1 Standard Analytical Bias

LabCode	STDGradeRangeBin	Samples	Average Z
BureauVeritas	0.10-1.00	20	0.06
BureauVeritas	1.00-2.00	17	0.16
BureauVeritas	2.00-4.00	12	-0.31
BureauVeritas	All	49	0.01
SGS_Bamako	0.10-1.00	53	-0.05
SGS_Bamako	1.00-2.00	41	0.04
SGS_Bamako	2.00-4.00	34	0.01
SGS_Bamako	>4.00	41	0.14
SGS_Bamako	All	169	0.03
SGS_Kayes	0.10-1.00	1334	-0.17
SGS_Kayes	1.00-2.00	879	-0.23
SGS_Kayes	2.00-4.00	574	0.12
SGS_Kayes	>4.00	143	0.54

LabCode	STDGradeRangeBin	Samples	Average Z
SGS_Kayes	All	2930	-0.10
SGS_Lakefield	0.10-1.00	3	0.00
SGS_Lakefield	1.00-2.00	7	0.26
SGS_Lakefield	>4.00	11	0.50
SGS_Lakefield	All	21	0.35
SGS_Morila	0.10-1.00	76	-0.58
SGS_Morila	1.00-2.00	59	-0.11
SGS_Morila	2.00-4.00	45	-0.73
SGS_Morila	All	180	-0.46
Combined	All	3349	-0.11

Core and RC Field Duplicates (FDUP)

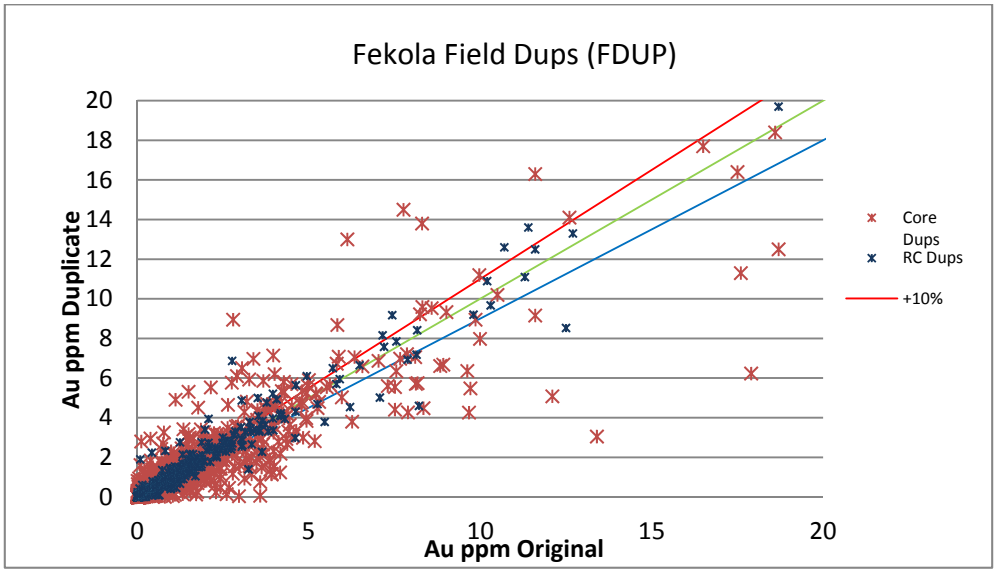
There are 2,987 core and 1,830 RC field duplicate pairs (FDUP) related to original samples contributing to the current resource update.

The total grade sum of the original assays (2,635.14 ppm) compares well with the total grade sum of the field duplicate assays (2,608.54 ppm) (Table 12.2.2). The total grade sum of the original core assays (1,766.91 ppm) compares well with the total grade sum of the core field duplicate assays (1,733.93 ppm). The total grade sum of the original RC assays (868.23 ppm) compares well with the total grade sum of the RC field duplicate assays (874.60 ppm).

Table 12.2.2 Field Duplicate Grade Sums

Sample Type	Population	Original Au Sum	Dup Au Sum	Orig/Dup%
RC	1830	868.23	874.60	99.3
Core	2987	1766.91	1733.93	101.9
All	4817	2635.14	2608.54	101.0

Figure 12.2.6 Fekola Field Duplicate Scatterplot



No significant bias is apparent in the scatterplot of the field duplicate data.

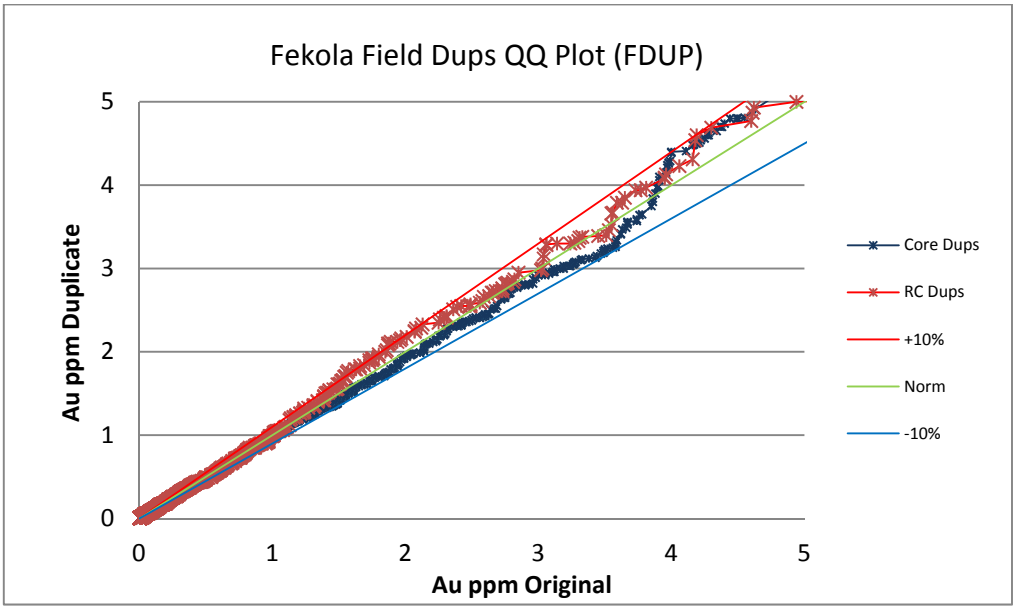
A comparison of the grade sums by grade range bin is presented in Table 12.2.3. The samples have been binned on the basis of original grade.

Table 12.2.3 Field Duplicate Grade Sums by Grade Range Bin

Sample Type	Grade Bin (g/t)	Population	Original Au Sum	Dup Au Sum	Orig/Dup%
Core	<0.25	2083	98.2	134.5	73.0
Core	0.25-0.50	254	91.9	99.8	92.0
Core	0.50-1.00	236	169.2	175.3	96.5
Core	1.00-2.50	226	357.3	348.3	102.6
Core	>2.50	188	1050.4	976.0	107.6
RC	<0.25	1373	67.8	72.4	93.6
RC	0.25-0.50	152	55.7	56.9	97.9
RC	0.50-1.00	119	87.1	86.9	100.3
RC	1.00-2.50	105	162.4	168.8	96.2
RC	>2.50	81	495.3	489.7	101.1
Sums / Avgs	**All**	**4817**	**2635**	**2609**	**101.0**

A QQ Plot of the field duplicate data shows reasonably neutral behaviour of both core and RC duplicates (Figure 12.2.7). There is a slight (~3.8%) positive bias to the RC duplicates between 1.0 and 2.5 ppm. It should be noted that the removal of three sample pairs, with relatively high absolute differences, from the limited sample population within this grade bin would eliminate this bias.

Figure 12.2.7 Fekola Field Dups QQ Plot



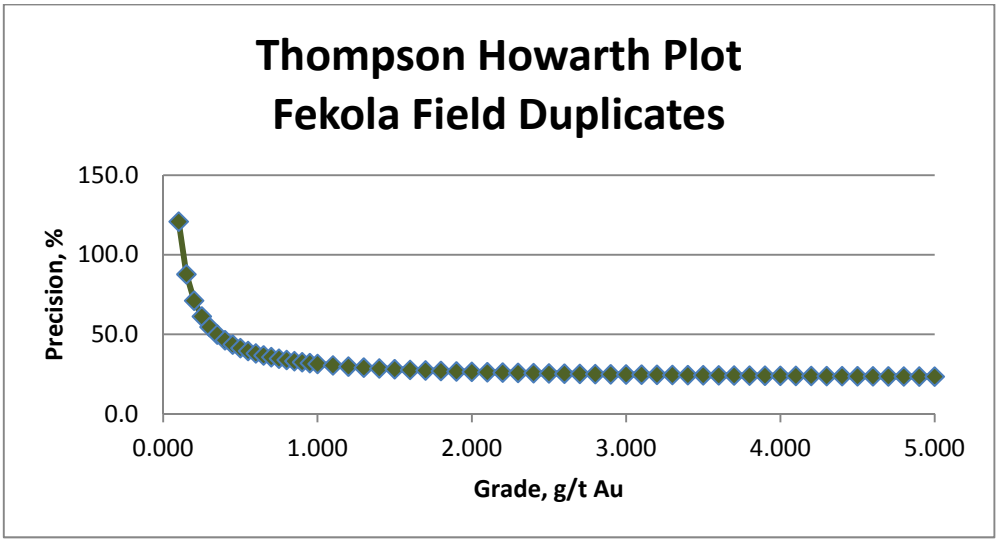
There is a 7% negative bias to the core duplicates between 2.5 ppm and 4 ppm. Removal of seven sample pairs, with relatively high absolute differences, from the limited sample population within this grade bin would minimize this bias.

The field dup population has been evaluated spatially and with respect to time. No distinct pattern is apparent that might suggest that samples from a specific latitude or depth or particular drill program are more prone to show a bias than another.

The Thompson-Howarth plot of Field Duplicate Precision (Figure12.2.8) shows that the observed bias is well within the limits of the calculated precision.

The overall field duplicate correlation is good.

Figure 12.2.8 Fekola Field Dups Thompson-Howarth Plot



Umpire Assays

There are 1,523 (1,454 excluding standards) Umpire assays, which is about 1.5% (~1 in 66 samples) of the original sample population contributing to the current resource update.

The Umpire population contains analyses from two umpire labs evaluating four primary labs. Three of the populations are rather small, reflecting their relative frequency in the population contributing to the current resource update.

Table 12.2.4 Umpire Population by Lab

OrigLabCode	UmpLabCode	Umpire Assays
BureauVeritas	SGS_Kayes	52
SGS_Bamako	BureauVeritas	49
SGS_Kayes	BureauVeritas	1,187
SGS_Morila	BureauVeritas	166
Sum		**1,454**

Table 12.2.5 Umpire Population Grade Sums by Lab

OrigLab	UmpLab	OrigAuSum	UmpAuSum	Orig/UMP%
BureauVeritas	SGS_Kayes	94.8	110.4	85.9
SGS_Bamako	BureauVeritas	67.7	61.4	110.3
SGS_Kayes	BureauVeritas	1879.2	1934.4	97.1
SGS_Morila	BureauVeritas	197.9	204.2	96.9
Sums		**2239.6**	**2310.4**	**96.9**

The total grade sums show a tendency for the Umpire assay to be higher than the original sample with the exception of the SGS_Bamako originals versus the Bureau Veritas Umpires (Table 12.2.5).

A few isolated outliers with poor correlation have a significant effect on the grade sums of the small SGS_Kayes versus Bureau Veritas and Bureau Veritas versus SGS_Bamako population sets (Figure 12.2.9). The general correlation is good with a few outliers that may represent sample mix-ups.

Figure 12.2.9 Fekola Umpires: Kayes vs BV, BV vs Bamako

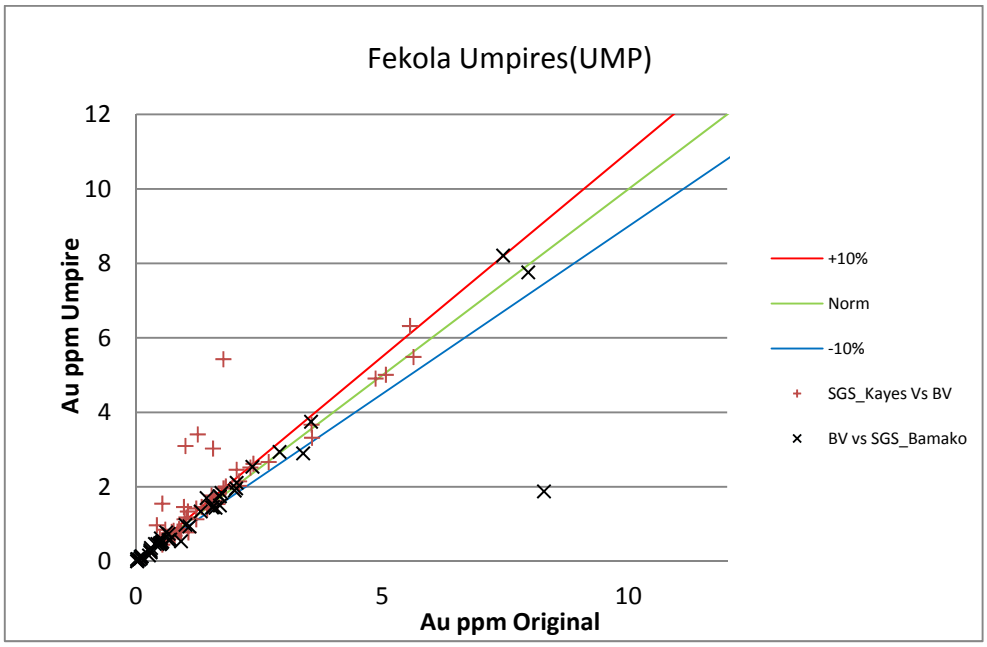
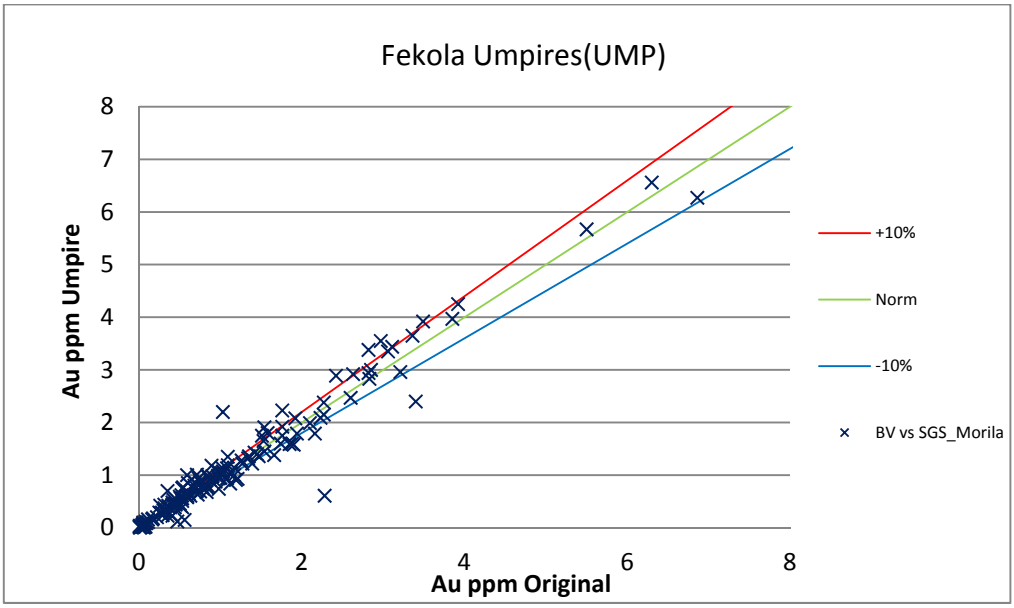
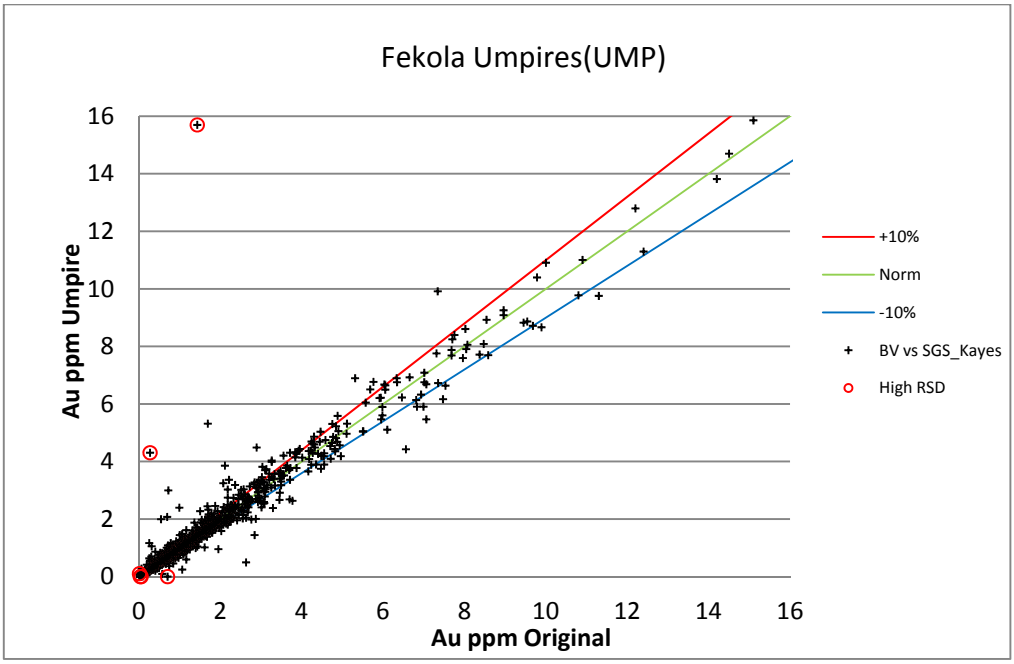


Figure 12.2.10 Fekola Umpires: BV vs Morila



The Bureau Veritas umpires of the SGS_Morila originals show a strong positive correlation with a slight positive bias to the umpires (Figure 12.2.10).

Figure 12.2.11 Fekola Umpires: BV vs Kayes



The Bureau Veritas umpires of the SGS_Kayes originals show a good general correlation. Six assay pairs with relative standard deviation (RSD) greater than 100%, possibly representing sample mix-ups, have been removed from the population presented in Figure 12.2.11 for subsequent evaluation.

Table 12.2.6 SGS_Kayes vs Bureau Veritas Umpire Grade Sums

Check Grade Bin	SumOf Orig Au SGS_Kayes	SumOf Ump Au Bureau Veritas	Orig/Ump%
<0.25	15.60	19.72	79.1
0.25 - 0.50	72.86	79.46	91.7
0.50 - 1.00	222.10	234.22	94.8
1.00 - 2.50	507.69	516.75	98.2
>2.50	1058.42	1064.15	99.5
Sums/Avgs	**1876.67**	**1914.30**	**98.0**

Removing the six high RSD assay pairs has decreased the overall Original vs. Umpire ratio from 97.1 to 98% (Table 12.2.6). The Umpires are biased high by 2% relative to the originals. This same sense of bias is also apparent in the standard analyses for the Bureau Veritas Umpire results when compared to the SGS_Kayes standard bias results (Table 12.2.7).

Table 12.2.7 Bureau Veritas vs SGS_Kayes Bias

STDGradeRangeBin	Standards	Average Z BV Umpire Std	Average Z Orig SGS Kayes
0.10 - 1.00	11	0.24	-0.17
1.0 2.00	23	0.50	-0.23
2.00 - 4.00	19	0.59	0.12
>4.00	6	1.84	0.54
All	**59**	**0.62**	**-0.10**

Evaluation of the standard bias of the umpire set versus the overall standard performance of the original assays suggests that the umpires are biased high relative to the originals.

The umpire assays provide good support for the original assays.

12.2.3 Comments on Section 12

The data verification completed by B2Gold was performed under the supervision of the QP. The QP considers that a reasonable level of verification has been completed during the work conducted to date, and that no material issues would have been left unidentified from the verification programs undertaken. The QP is of the opinion that the data verification programs undertaken on the data collected from the Project adequately reflect deposit dimensions, true widths of mineralization, and

the style of the deposits, and adequately support the geological interpretations, the analytical and database quality.

No problems with the database, sampling protocols, flowsheets, check analysis program, or data storage were identified that were sufficient to preclude the use of the database for estimation purposes. The quality of the gold analytical data is considered sufficiently reliable to support Mineral Resource estimation without limitations on Mineral Resource confidence categories. The data can be used in mine planning.

FEKOLA GOLD PROJECT

NI 43-101 - MINERAL PROCESSING AND METALLURGICAL TESTING

3174-STY-002

Table of Contents

			Page
13.0	**MINERAL PROCESSING AND METALLURGICAL TESTING**		**13.1**
	13.1	Introduction	13.1
	13.2	Metallurgical Samples	13.2
		13.2.1 Geology	13.2
		13.2.2 Metallurgical Domains	13.4
		13.2.3 Metallurgical Samples	13.6
		13.2.4 Comminution Samples	13.8
	13.3	Metallurgical Testing	13.8
		13.3.1 Metallurgical Testwork Programme	13.8
		13.3.2 Head Analysis	13.13
		13.3.3 Mineralogy	13.19
		13.3.4 Comminution Testwork	13.20
		13.3.5 Gravity Concentration	13.22
		13.3.6 Grind / Recovery Testwork	13.23
		13.3.7 Preg Robbing Assessment	13.24
		13.3.8 Whole Ore Leach Cyanidation Optimisation Testwork	13.25
		13.3.9 Whole Ore Cyanidation of Variability Samples at Optimised Leach Conditions	13.33
		13.3.10 Diagnostic Leaching Testwork	13.36
		13.3.11 Bulk Cyanidation Tests	13.37
		13.3.12 Cyanide Destruction Testwork	13.37
		13.3.13 Oxygen Uptake Testwork	13.38
		13.3.14 Carbon Modelling	13.40
		13.3.15 Slurry Rheology Testwork	13.43
		13.3.16 Thickening and Flocculation Testwork	13.44
		13.3.17 Materials Handling Testwork	13.45
	13.4	Comminution Circuit Selection	13.46
		13.4.1 Comminution Circuit Design Basis	13.46
		13.4.2 Comminution Circuit Modelling	13.46
		13.4.3 Comminution Circuit Selection	13.46
	13.5	Metallurgical Testwork Summary	13.47
		13.5.1 Selected Treatment Route	13.47
		13.5.2 Key Process Design Parameters	13.47
		13.5.3 Metallurgical Recoveries	13.48
		13.5.4 Reagent Consumption	13.50
		13.5.5 Testwork Conclusions	13.50

TABLES

Table 13.2.1	Met Domains Gold Head Grade	13.4
Table 13.2.2	Fekola Deposit – Resource Tonnage Distribution across Met Domains	13.6
Table 13.2.3	Fekola Deposit – Resource Grade Distribution across Met Domains	13.6
Table 13.3.1	Summary of Metallurgical Testwork Programme	13.9
Table 13.3.2	Metallurgical Domain Composite Head Analysis	13.14
Table 13.3.3	Metallurgical Domain Composite Head Analysis (Part 1 of 3)	13.16
Table 13.3.4	Metallurgical Domain Composite Head Analysis (Part 2 of 3)	13.17
Table 13.3.5	Metallurgical Domain Composite Head Analysis (Part 3 of 3)	13.18
Table 13.3.6	Bulk Mineralogy by QEMSCAN	13.19
Table 13.3.9	Gravity Separation Summary by SGS Lakefield	13.22

Table of Contents (Continued)

Page

Table 13.3.10	Gravity Separation Summary by Dawson Met Lab	13.23
Table 13.3.11	Summary of Grind / Recovery Leach Results of Whole Ore Cyanidation	13.24
Table 13.3.12	Summary of Whole Ore vs CIL Cyanidation	13.25
Table 13.3.13	Summary of Whole Ore Cyanidation of Variability Samples at Optimised Leach Conditions	13.34
Table 13.3.14	Summary of Bulk Whole Ore Cyanidation Tests	13.37
Table 13.3.15	Carbon Adsorption of High Grade Domain Composite Sample	13.41
Table 13.3.16	Summary of Fekola Slurry Rheology	13.43
Table 13.3.17	Summary of Thickening Testwork Results	13.44
Table 13.5.1	Summary of Grinding, Leach and CIP Circuit Design Parameters	13.48
Table 13.5.2	Summary of Lime and Sodium Cyanide Addition	13.50

FIGURES

Figure 13.2.1	Plan map of Fekola Deposit Geology	13.3
Figure 13.2.2	Plan View Location of Metallurgical Drillholes within Stage 7 Design Pit	13.5
Figure 13.2.3	Spatial Location of Metallurgical Drillholes within Stage 7 Design Pit	13.5
Figure 13.2.4	Longsection of Metallurgical Samples Origin	13.7
Figure 13.2.5	Longsection of Comminution Samples Origin	13.8
Figure 13.3.1	Metallurgical Testwork Programme Flow Chart	13.12
Figure 13.3.2	Effect of Residence Time	13.26
Figure 13.3.3	Effect of Lead Nitrate Addition	13.28
Figure 13.3.4	Effect of Cyanide Concentration	13.29
Figure 13.3.5	Effect of Dissolved Oxygen	13.30
Figure 13.3.6	Effect of Pulp Density	13.31
Figure 13.3.7	Reproducibility of Optimum Gold Leaching Conditions	13.32
Figure 13.3.8	Oxygen Demand of High Grade Domain Composite Sample	13.39
Figure 13.3.9	Oxygen Demand of Low Grade Domain Composite Sample	13.39
Figure 13.3.10	Oxygen Demand of Hanging Wall Domain Composite Sample	13.40
Figure 13.3.11	Carbon Adsorption of Bulk Leached HG Composite Sample	13.41
Figure 13.3.12	Equilibrium Loading of Gold onto Carbon from Bulk Leached HG Composite Sample	13.42
Figure 13.5.1	Gold Extraction Model	13.49

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A metallurgical testwork programme has been completed to support the OFS of the Project. The majority of the programme was conducted by SGS Canada Inc. ('SGS Lakefield') in Lakefield, Ontario, Canada between 2014 and 2015. SGS Lakefield is accredited to the requirements of ISO/IEC 17025 for relevant mineralogical, geochemical and trade mineral tests conducted for the study. The remainder of the program was either completed at SGS Lakefield under supervision of a consultant or specialist or at another laboratory facility during the same time period. The testwork programme and laboratory details are listed in Table 13.3.1.

The programme included testwork to establish:

- Comminution characteristics of the ore, three comminution domain composites and eighteen comminution variability samples.
- Materials handling properties for three comminution domain composites.
- Mineralogy for three metallurgical domain composites.
- Head analysis of three metallurgical domain composites and eighteen metallurgical variability samples.
- Gravity recoverable gold content of three metallurgical domain composites.
- Optimum grind sizes for whole ore cyanidation leach of three metallurgical domain composites.
- Comparison of whole ore and CIL cyanidation leach of three metallurgical domain composites.
- Optimum whole ore cyanidation leach conditions for the high grade domain composite.
- Leach performance of eighteen metallurgical variability samples.
- Diagnostic leach analysis of two high grade samples.
- Cyanide destruction testwork for the high grade domain composite after optimum whole ore cyanidation leach conditions.
- Engineering data, including testing of oxygen uptake, slurry rheology, carbon kinetics and thickener settling rate.

This section summarises the metallurgical testwork and draws conclusions relevant for process plant design criteria.

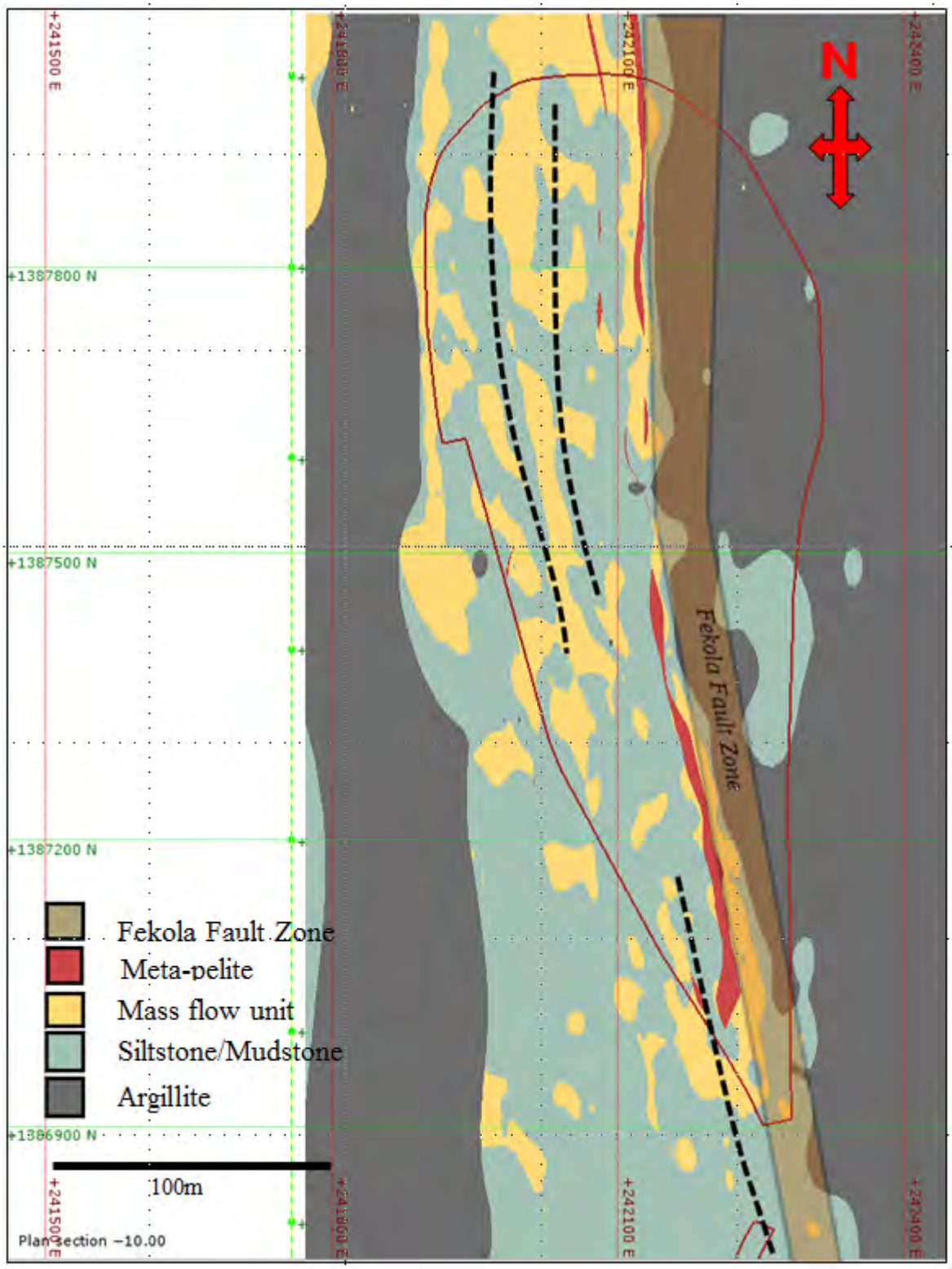
13.2 Metallurgical Samples

13.2.1 Geology

The Fekola deposit is considered to be an example of an orogenic-style gold deposit. Orogenic gold deposits dominantly form in metamorphic rocks in the mid to shallow crust, at or above the brittle-ductile transition, in compressional settings that facilitate transfer of hot gold-bearing fluids from deeper levels. During a geological event, the rapid rise of fluids takes these out of equilibrium with their surroundings, resulting in gold precipitation. The geological setting for the formation of the orogenic-style gold deposits at Fekola is described in more detail in Section 8 of this report.

The Fekola deposit is hosted by a moderate to steeply west dipping, folded sequence of marine meta-sediments of the Kofi group, which locally include: argillite and minor pelitic sediments; fine grained turbidites, comprising laminated to thin-bedded siliciclastic siltstone and mudstone; and a heterolithic, mass flow breccia, or conglomerate, as shown in Figure 13.2.1 and described in Section 7 of this report. Fine grained thin marble units are present as a volumetrically minor rock type interbedded with the politic units. Minor mafic volcanic, or volcaniclastic units occur locally in the upper and possibly, lower portions of the hanging wall stratigraphy. Weakly feldspar-phyric felsic dykes are locally observed. The deposit has been subjected to upper greenschist facies metamorphism.

Figure 13.2.1 Plan map of Fekola Deposit Geology



As shown in Figure 13.2.1., deposit area is underlain by west dipping politic rocks. Late-stage Fekola Fault Zone (shown in brown) is parallel in strike to mineralization. Proposed pit outline shown in red. Dashed lines indicate inferred trace of D2 structures as detailed in Section 7 of this report. Scale as indicated by UTM grid lines.

- Mineralisation is broadly continuous and has been traced over a strike extent of approximately 1.5 km, to depths of up to 400 m below surface, with widths to 300 m. The greatest continuity is observed within a high grade shoot (>2 g/t Au) which plunges approximately 13° to the north-north west. Mineralisation is open at depth, down plunge.

The majority of gold mineralisation in the Fekola deposit occurs in unweathered, fresh rock and is preferentially associated with stringers of pyrite parallel to the foliation and in fine disseminated pyrite. More specifically, the gold mineralisation is associated with fine grained pyrite in tan dolomite-albite altered wall rocks, which locally contain diffuse, often deformed pale grey quartz-dolomite-pyrite-albite matrix veins and veinlets. Trace amounts of chalcopyrite have also been observed. The total sulphide content of the deposit is typically less than 5%.

13.2.2 Metallurgical Domains

Due to the orogenic-style of the deposit, the gold mineralisation is not limited to a particular lithology, but is proximally associated with alterations throughout the deposit. The domains for the metallurgical testwork programme have been classified according to the gold head grades along with a spatial component and consideration for the alteration intensity in the selection process. The three domains identified for the Fekola deposit are High Grade Domain, Hanging Wall Domain, which is of medium gold grade, and Low Grade Domain.

Table 13.2.1 Met Domains Gold Head Grade

Domain	Unit	Gold Grade
High Grade	g/t	2.0 – 5.0
Hanging Wall	g/t	1.0 – 2.0
Low Grade	g/t	0.6 – 1.0

For the metallurgical testwork programme, five drillholes were identified and drilled in July 2014. A total of 1,271 metres of HQ-diameter diamond drill core was retrieved from the five metallurgical drillholes, FK_MET01 to FK_MET05. The locations of the five metallurgical drillholes are shown in Figure 13.2.1 and Figure 13.2.2.

Figure 13.2.2 Plan View Location of Metallurgical Drillholes within Stage 7 Design Pit

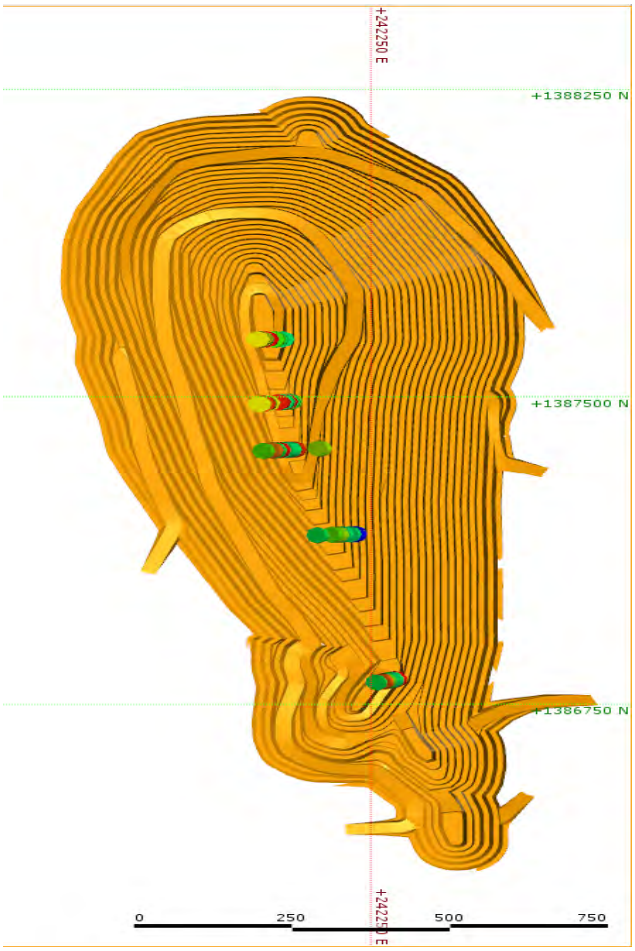
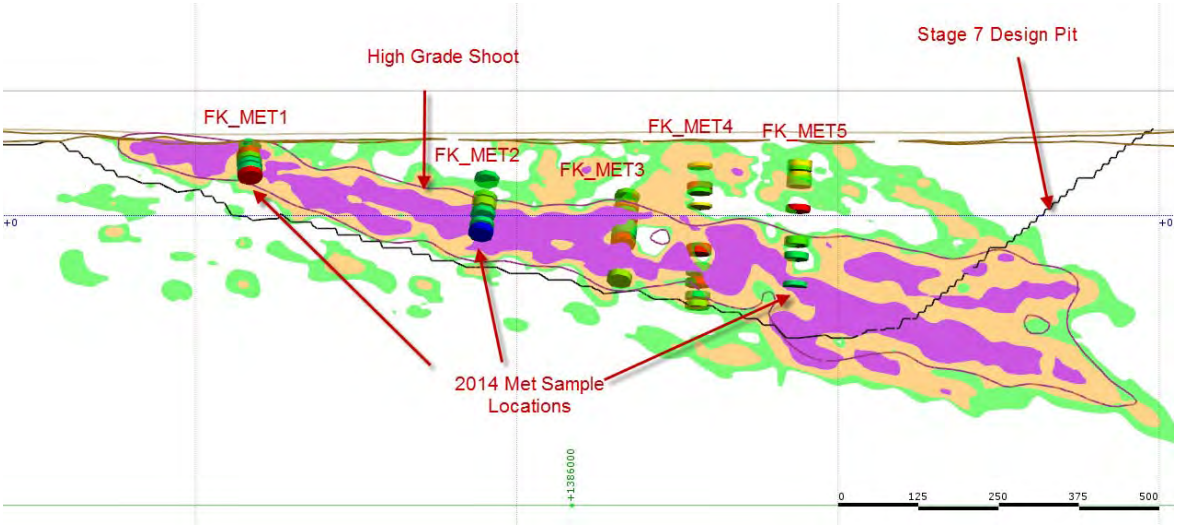


Figure 13.2.3 Spatial Location of Metallurgical Drillholes within Stage 7 Design Pit



The three metallurgical domains have since been correlated with the latest mineral resource model released in early 2015. The distribution of the metallurgical domains have been based on a mineral resource with a cut-off grade of 0.6 g/t inside the Stage 7 pit with ore inside and outside the high grade shoot. The estimated distribution by tonnage of these domains for the metallurgical testwork program is shown in Table 13.2.3. The average grade for each metallurgical domain is shown in Table 13.2.3.

Table 13.2.2 Fekola Deposit – Resource Tonnage Distribution across Met Domains

Domain	Tonnes	Distribution,%
High Grade	13,597,264	23.1
Hanging Wall	30,153,754	51.2
Low Grade	15,123,843	25.7
Subtotal	**58,874,860**	**100.0**

Table 13.2.3 Fekola Deposit – Resource Grade Distribution across Met Domains

Ore Type	Unit	Grade
High Grade	g/t	5.23
Hanging Wall	g/t	1.56
Low Grade	g/t	0.76
Subtotal		**2.20**

13.2.3 Metallurgical Samples

The metallurgical testwork programme is based on three metallurgical domain samples and eighteen metallurgical variability samples. The samples for the comminution part of the testwork programme were compiled separately and are described in Section 13.2.4.

The three metallurgical domain samples, High Grade (HG), Low Grade (LG) and Hanging Wall (HW), have been compiled with HQ-diameter half core from all five metallurgical drillholes, FK_MET01 to FK_MET05 inclusive. The sample selection for these domain composites was based on achieving desired gold grade, spatial representation and alteration intensity. The final metallurgical domain sample weights were 250 kg of HG and about 120 kg each of LG and HW.

The eighteen metallurgical variability samples were sourced from the HQ-diameter half core from the five metallurgical drillholes, FK_MET01 to FK_MET05. Each variability sample is a contiguous length with seven representing HG, seven for LG and four for HW. Each variability sample was selected based on geologic variability, such as deep or shallow spatial location, or as a strong albite-carbonate-sericite-pyrite alteration or a high pyrite mineralisation zone. The final metallurgical variability sample weights were in the range of 30 to 50 kg each.

Ten of the metallurgical variability and comminution variability samples are common and cover the same length of a drillhole. The drillholes extend across the pit along the pit centre-line and the origin of each metallurgical domain and variability sample is detailed in Figure 13.2.4.

Figure 13.2.4 Longsection of Metallurgical Samples Origin

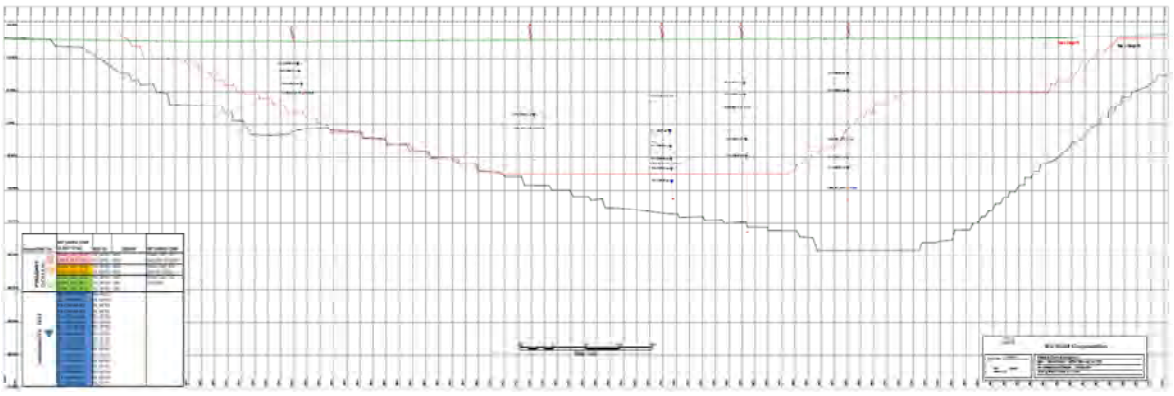
13.2.4 Comminution Samples

The comminution testwork programme is based on three comminution domain samples and eighteen comminution variability samples. The three comminution domain samples, Shallow High Grade (Shallow HG), Deep High Grade (Deep HG) and Low Grade (LG), have been compiled with HQ-diameter whole core from the five metallurgical drillholes, FK_MET01 to FK_MET05 inclusive. The sample selection for these domain composites was based on spatial representation and alteration intensity. As these samples are whole drill core, the gold grade wasn't able to be measured. The three comminution domain samples weighed approximately 220 kg each.

The eighteen comminution variability samples were sourced from the HQ-diameter half core from the five metallurgical drillholes, FK_MET01 to FK_MET05. Each variability sample is a contiguous length with ten representing Deep HG and eight representing Shallow HG and LG. Each variability sample was selected based on geologic variability, such as deep or shallow spatial location and extent of mineralisation and alteration intensities. The final comminution variability sample weights were in the range of 40 to 50 kg each.

Ten of the comminution variability and metallurgical variability samples are common and cover the same length of a drillhole. The drillholes extend across the pit along the pit centre-line and the origin of each comminution domain and variability sample is detailed in Figure 13.2.5.

Figure 13.2.5 Longsection of Comminution Samples Origin



13.3 Metallurgical Testing

13.3.1 Metallurgical Testwork Programme

The metallurgical testwork programme was completed between 2014 and 2015. The majority of the programme was conducted by SGS Lakefield. The remainder of the programme was completed by either SGS Lakefield under supervision of a consultant or specialist or at another laboratory facility. The testwork programme and laboratory details are listed in Table 13.3.1. This information is also shown diagrammatically in Figure 13.3.1.

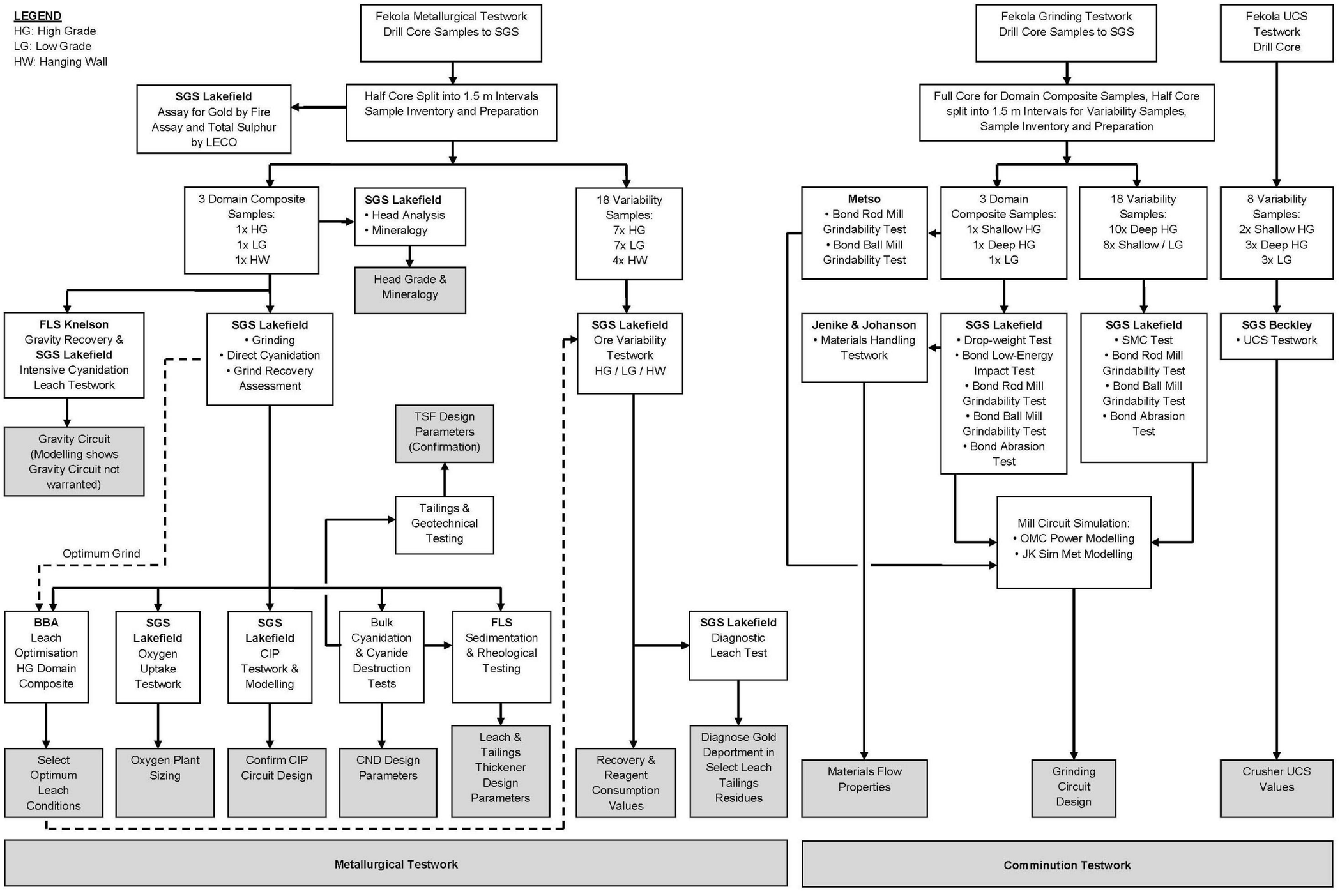
Table 13.3.1 Summary of Metallurgical Testwork Programme

Testwork Description	Laboratory	Samples	Scope
Materials Handling Testwork	Jenike & Johanson	*3 x Domain Composites:* 1 x Shallow HG 1 x Deep HG 1 x LG	Particle density determination Compressibility tests Loose and compacted bulk density tests Flow function tests Wall friction determination Critical chute angle determination
Comminution Testwork	SGS Lakefield	*3 x Domain Composites:* 1 x Shallow HG 1 x Deep HG 1 x LG *18 x Variability Samples:* 10 x Deep HG 8 x Shallow HG / LG	JK drop-weight test Bond low-energy impact test Bond rod mill grindability test Bond ball mill grindability test Bond abrasion test SMC test Bond rod mill grindability test Bond ball mill grindability test Bond abrasion test
Comminution Testwork Comparison	Metso	*3 x Domain Composites*: 1 x Shallow HG 1 x Deep HG 1 x LG	Bond rod mill grindability test Bond ball mill grindability test
UCS Testwork	SGS Beckley	*8 x Variability Samples:* 2 x Shallow HG 3 x Deep HG 3 x LG	Unconfined (uni-axial) compressive strength with elastic modulus (UCM)
Chemical Composition and Mineralogy	SGS Lakefield	*3 x Domain Composites:* 1 x HG 1 x LG 1 x HW	Comprehensive chemical analysis and assay work Gold deportment study QEMSCAN and XRD study

Testwork Description	Laboratory	Samples	Scope
Gravity Concentration Testwork	SGS Lakefield Dawson Met lab	*3 x Domain Composites:* 1 x HG 1 x LG 1 x HW	Gravity concentration
Grind / Recovery Testwork	SGS Lakefield	*3 x Domain Composites:* 1 x HG 1 x LG 1 x HW	Grind and recovery leach testwork for optimum grind size determination
Leach Optimisation Testwork	Managed by BBA and conducted at Process Research Ortech	*1 x Domain Composite:* 1 x HG	Leach optimisation testwork on HG to evaluate the effect of each of the following parameters with respect to gold recovery and leach kinetics: • Residence time (24hrs, 36hrs, 48hrs) • Lead nitrate addition (50g/t, 100g/t, 250g/t) • Cyanide concentration (150ppm, 250ppm, 350ppm, 450ppm, 550ppm) • Dissolved oxygen level (8ppm, 15ppm) • Pulp density (40%, 45%, 50% solids)
Leach Variability Testwork	SGS Lakefield	*18 x Variability Samples:* 7 x HG 7 x LG 4 x HW	Sample preparation and head analysis Direct cyanidation with optimum leach conditions Reagent consumption rate determination Diagnostic leaching testing on select samples
Cyanide Destruction Testwork	SGS Lakefield	1 x HG Bulk Sample after Cyanide Leach	Batch SO^2/air cyanide destruction tests Continuous SO^2/air cyanide destruction tests
Oxygen Uptake Testwork	SGS Lakefield	*3 x Domain Composites:* 1 x HG 1 x LG 1 x HW	Oxygen uptake rate determination
Carbon Adsorption Testwork	SGS Lakefield	*3 x Domain Composites:* 1 x HG 1 x LG 1 x HW	Leach and carbon adsorption kinetic tests Determination of carbon loading isotherm

Testwork Description	Laboratory	Samples	Scope
Sedimentation and Rheological Testwork	SGS Lakefield FLSmidth	*3 x Domain Composites:* 1 x HG 1 x LG 1 x HW + 1 x HG CND Tailings Sample	Bench scale sedimentation tests that included the following: Flocculant screening Determination of optimum feed solids dilution settling tests Thickener sizing Thickener underflow rheology measurements
Environmental and Geotechnical Testing ***(This testwork will be conducted and reported in the future)***	SGS Lakefield	1 x HG Bulk Sample after Cyanide Leach and Cyanide Destruction	Comprehensive solution and solids analyses Modified acid base accounting Net acid generation testing De-ionised water leach TCLP testing Whole rock analyses Specific gravity determination Determination of particle size distribution Settling tests Drained settling tests Standard proctor tests Determination of Atterberg limits Air drying tests Rowe cell consolidation with hydraulic conductivity triaxial tests

Figure 13.3.1 Metallurgical Testwork Programme Flow Chart



13.3.2 Head Analysis

Metallurgical Domain Composite Samples

The metallurgical domain composites (High Grade, Low Grade and Hanging Wall) were assayed for the following:

- Duplicate Au by 30 g fire assay, Ag by AA, S_T, $S^{=}$, carbon speciation (C_T, C_G, C_{ORG}, CO_2), As, Bi, Cu, Hg, Zn, Se, Te, ICP scan and cyanide soluble gold by ALS method.

The samples were also submitted for a specific gravity measurement (by pycnometer). The results are summarised in Table 13.3.2.

Table 13.3.2 Metallurgical Domain Composite Head Analysis

Element	Unit	High Grade Domain FEK-MET-001	Low Grade Domain FEK-MET-002	Hanging Wall Domain FEK-MET-003
Au Cut A	g/t	4.15	0.84	1.87
Au Cut B	g/t	4.30	0.81	1.82
Au Avg.	***g/t***	***4.23***	***0.83***	***1.85***
Au (calc.)	*g/t*	*4.48*	*0.83*	*1.78*
Ag	g/t	< 0.5	< 0.5	< 0.5
AuCN	g/t	3.7	0.6	1.5
S_T	%	1.97	1.11	1.14
$S^=$	%	1.87	0.96	1.12
C_T	%	3.33	3.21	2.94
C_G	%	0.03	0.02	0.03
C_{ORG}	%	0.17	0.12	0.11
CO_2	%	12.0	12.0	11.0
As	%	<0.001	<0.001	<0.001
Bi	%	<0.002	<0.002	<0.002
Cu	%	0.005	<0.002	0.003
Hg	%	<0.3	<0.3	<0.3
Zn	%	0.003	0.002	0.002
Se	g/t	<10	<10	<10
Te	g/t	<4	7.00	<4
SG		2.91	2.86	2.88
ICP Scan				
Al	g/t	59800	62900	67400
As	g/t	< 30	< 30	< 30
Ba	g/t	202	391	195
Be	g/t	0.94	0.92	0.86
Bi	g/t	< 20	< 20	< 20
Ca	g/t	69100	62000	60400
Cd	g/t	< 2	< 2	< 2
Co	g/t	24	21	15
Cr	g/t	147	124	102
Cu	g/t	68	10	19
Fe	g/t	45000	43600	37800
K	g/t	6670	7640	10300
Li	g/t	8	8	7
Mg	g/t	41700	39400	34700
Mn	g/t	644	568	492
Mo	g/t	< 5	< 5	< 5
Na	g/t	38800	36700	39700
Ni	g/t	85	75	59
P	g/t	711	649	670
Pb	g/t	28	< 20	< 20
Sb	g/t	< 10	< 10	< 10
Se	g/t	< 30	< 30	< 30
Sn	g/t	< 20	< 20	< 20
Sr	g/t	197	146	136
Ti	g/t	2890	2630	2880
Tl	g/t	< 30	< 30	< 30
U	g/t	< 20	< 20	< 20
V	g/t	110	113	106
Y	g/t	12.6	11.7	12.6
Zn	g/t	53	28	25

Au (calc.) = calculated average from tests completed during program

Observations include:

- Gold head grades ranged from 0.83 g/t to 4.23 g/t for the three metallurgical domain composite samples.
- Silver head grades were all less than 0.5 g/t.
- Sulphide grades ranged from 0.96% to 1.87%.

Metallurgical Variability Samples

The eighteen metallurgical variability samples were assayed for the following:

- Duplicate Au by 30 g fire assay, Ag by AA, S_T, $S^{=}$, C_G, C_{ORG}, ICP scan, and cyanide soluble gold by ALS method.

A summary of the head analysis is shown in Tables 13.3.3, 13.3.4, and 13.3.5.

Table 13.3.3 Metallurgical Domain Composite Head Analysis (Part 1 of 3)

Element	Unit	FEK-MET-004	FEK-MET-005	FEK-MET-006	FEK-MET-007	FEK-MET-008	FEK-MET-009
Domain		High	High	High	High	High	High
Au Cut A	g/t	11.1	12.3	4.23	4.33	3.53	4.37
Au Cut B	g/t	11.2	13.1	4.00	4.97	3.59	7.38
Au Avg.	***g/t***	***11.2***	***12.7***	***4.12***	***4.65***	***3.56***	***5.88***
Ag	g/t	0.7	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
AuCN	g/t	9.4	10.7	4.3	4.2	2.8	5.1
S_T	%	2.81	1.06	1.33	2.33	2.88	1.22
$S^=$	%	2.64	0.95	1.31	2.28	2.77	1.11
C_G	%	0.02	0.04	0.03	0.04	0.04	0.09
C_{ORG}	%	0.18	0.26	0.16	0.08	0.12	0.09
ICP Scan							
Al	g/t	55500	63800	63600	61800	56000	66500
As	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Ba	g/t	64.4	77	229	104	29.7	111
Be	g/t	0.52	0.68	0.60	0.58	0.93	0.60
Bi	g/t	< 20	< 20	< 20	< 20	< 20	< 20
Ca	g/t	78000	77500	64300	67700	70300	66100
Cd	g/t	< 2	< 2	< 2	< 2	< 2	< 2
Co	g/t	31	17	16	31	33	11
Cr	g/t	201	72	129	197	124	94
Cu	g/t	515	61.0	58.4	9.3	49.6	28.0
Fe	g/t	48900	38800	38000	36100	57000	28400
K	g/t	1820	2150	6330	1630	3170	3670
Li	g/t	< 5	< 5	< 5	< 5	31	< 5
Mg	g/t	34900	39200	36300	34300	47800	35400
Mn	g/t	853	686	458	502	828	435
Mo	g/t	< 5	< 5	< 5	10	< 5	< 5
Na	g/t	41800	47100	44100	48900	31100	49100
Ni	g/t	73	57	65	61	91	49
P	g/t	764	670	821	740	837	681
Pb	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Sb	g/t	< 20	< 20	< 20	< 20	< 20	< 20
Se	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Sn	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Sr	g/t	287	262	127	219	172	226
Ti	g/t	2350	3210	3100	2840	2920	2860
Tl	g/t	< 30	< 30	< 30	< 30	< 30	< 30
U	g/t	< 20	< 20	< 20	< 20	< 20	< 20
V	g/t	71	106	84	74	84	92
Y	g/t	11.1	12.2	13.4	10.5	13.7	12.3
Zn	g/t	92	24	17	19	29	19

Table 13.3.4 Metallurgical Domain Composite Head Analysis (Part 2 of 3)

Element	Unit	FEK-MET-010	FEK-MET-011	FEK-MET-012	FEK-MET-013	FEK-MET-014	FEK-MET-015
Domain		High	HW	HW	HW	HW	LOW
Au Cut A	g/t	2.78	1.70	1.27	2.41	2.35	0.60
Au Cut B	g/t	2.77	1.65	1.31	2.93	2.32	0.57
Au Avg.	***g/t***	***2.78***	***1.68***	***1.29***	***2.67***	***2.33***	***0.58***
Ag	g/t	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
AuCN	g/t	2.5	1.5	1.0	2.2	2.0	0.4
S_T	%	1.01	1.35	0.86	1.19	1.03	0.89
$S^=$	%	0.98	1.24	0.80	1.18	0.98	0.80
C_G	%	0.05	0.03	0.04	0.03	0.03	0.01
C_{ORG}	%	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
ICP Scan							
Al	g/t	62600	64800	68100	63900	63900	65700
As	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Ba	g/t	214	261	263	113	180	259
Be	g/t	0.98	1.06	1.21	0.56	0.82	0.80
Bi	g/t	< 20	< 20	< 20	< 20	< 20	< 20
Ca	g/t	66100	67200	63100	63400	65400	68400
Cd	g/t	< 2	< 2	< 2	< 2	< 2	< 2
Co	g/t	25	24	21	13	19	18
Cr	g/t	135	131	101	68	260	152
Cu	g/t	2.4	1.4	6.1	6.0	1.4	2.1
Fe	g/t	43800	47000	43000	28100	46500	39500
K	g/t	9400	14400	13500	2230	9570	6890
Li	g/t	< 5	6	9	< 5	< 5	< 5
Mg	g/t	38200	41000	36000	33900	44400	38700
Mn	g/t	591	567	602	431	556	479
Mo	g/t	< 5	< 5	< 5	< 5	< 5	< 5
Na	g/t	35200	29600	34300	49100	36800	37500
Ni	g/t	77	102	67	55	129	75
P	g/t	709	695	687	599	701	596
Pb	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Sb	g/t	< 20	< 20	< 20	< 20	< 20	< 20
Se	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Sn	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Sr	g/t	140	120	127	231	131	155
Ti	g/t	2630	2780	2600	2860	3090	2670
Tl	g/t	< 30	< 30	< 30	< 30	< 30	< 30
U	g/t	< 20	< 20	< 20	< 20	< 20	< 20
V	g/t	115	126	116	77	110	105
Y	g/t	13.3	13.2	14.3	12.2	11.7	13.1
Zn	g/t	22	24	22	14	15	16

Table 13.3.5 Metallurgical Domain Composite Head Analysis (Part 3 of 3)

Element	Unit	FEK-MET-016	FEK-MET-017	FEK-MET-018	FEK-MET-019	FEK-MET-020	FEK-MET-021
Domain		LOW	LOW	LOW	LOW	LOW	LOW
Au Cut A	g/t	1.03	1.49	1.11	0.79	0.89	0.88
Au Cut B	g/t	0.94	1.50	1.06	0.71	0.80	0.96
Au Avg.	***g/t***	***0.98***	***1.49***	***1.08***	***0.75***	***0.84***	***0.92***
Ag	g/t	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
AuCN	g/t	0.8	1.2	1.2	0.7	0.6	0.7
S_T	%	3.16	1.58	1.31	1.01	1.46	1.18
$S^=$	%	2.02	1.54	1.17	1.01	1.27	0.75
C_G	%	0.05	0.01	0.03	0.03	0.04	0.04
C_{ORG}	%	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	0.14
ICP Scan							
Al	g/t	58600	63700	66500	65100	71000	61300
As	g/t	< 30	< 30	< 30	< 30	< 30	...
Ba	g/t	79	91.5	137	255	426	96.7
Be	g/t	0.76	0.96	1.12	0.76	1.24	1.11
Bi	g/t	< 20	< 20	< 20	< 20	< 20	< 20
Ca	g/t	81600	62900	62800	64900	62900	66000
Cd	g/t	< 2	< 2	< 2	< 2	< 2	< 2
Co	g/t	29	24	16	12	13	34
Cr	g/t	68	180	84	55	63	...
Cu	g/t	40.5	6.9	2.9	44.8	20.6	13.2
Fe	g/t	44400	42500	39800	31300	35900	49200
K	g/t	5530	10000	11500	9510	16200	14200
Li	g/t	< 5	< 5	8	6	12	10
Mg	g/t	33000	36600	35500	35000	32300	39300
Mn	g/t	705	605	517	478	412	712
Mo	g/t	< 5	< 5	< 5	< 5	< 5	< 5
Na	g/t	37200	36200	35000	40500	34600	19700
Ni	g/t	49	63	58	38	47	57
P	g/t	512	615	594	626	583	590
Pb	g/t	< 30	< 30	< 30	< 30	< 30	< 20
Sb	g/t	< 20	< 20	< 20	< 20	< 20	< 10
Se	g/t	< 30	< 30	< 30	< 30	< 30	< 30
Sn	g/t	< 30	< 30	< 30	< 30	< 30	< 20
Sr	g/t	227	142	135	120	131	86.7
Ti	g/t	2250	2540	2650	2610	3030	1640
Tl	g/t	< 30	< 30	< 30	< 30	< 30	< 30
U	g/t	< 20	< 20	< 20	< 20	< 20	< 20
V	g/t	71	100	103	98	133	102
Y	g/t	9.8	12.1	13.3	11.5	13.0	12
Zn	g/t	12	18	26	13	11	< 20

Observations include:

- Gold head grades ranged from 0.58 g/t to 12.7 g/t for the eighteen metallurgical variability samples.
- Silver head grades were low and below 0.7 g/t, with the majority below the detection limit(<0.5 g/t).

- Sulphide head grades ranged from 0.75% to 2.77% and averaged 1.38%.

13.3.3 Mineralogy

Each metallurgical domain composite sample was submitted for a mineralogical study which included QEMSCAN analysis and visible gold deportment. Whole ore samples were ground to the target grind size P_{80} (75 µm) and submitted for the testwork.

Bulk Mineralogy

The bulk mineralogy was carried out by QEMSCAN using Particle Map Analysis. XRD analysis including clay mineral speciation was also performed. The results are listed in Table 13.3.6. The three metallurgical domain composite samples, HG, LG and HW, had a similar mineral assemblage that was comprised of major amounts of plagioclase (~31%), moderate amounts of quartz (12 to 14%) and dolomite (14 to 16%), minor amounts (2 to 10%) of calcite, amphibole / pyroxene, clays, micas, chlorite, pyrite, and trace amounts (<2%) of other minerals. The main sulphide mineral was pyrite (2 to 4%), with trace amounts (<0.1%) of chalcopyrite, sphalerite, galena, and other sulphides. The main clay minerals were illite-montmorillonite mixture, chlorite, and illite for all three samples.

Table 13.3.6 Bulk Mineralogy by QEMSCAN

Sample		High Grade Comp	Low Grade Comp	Hanging Wall Comp
Mineral Mass (%)	Pyrite	3.45	1.96	2.41
	Sphalerite	0.00	0.01	0.00
	Chalcopyrite	0.02	0.01	0.02
	Galena	0.00	0.01	0.00
	Other Sulphides	0.00	0.01	0.00
	Fe-Oxides	0.12	0.95	0.18
	Other Oxides	0.61	0.57	0.58
	Quartz	12.7	14.4	14.4
	K-Feldspar	0.79	0.77	1.57
	Plagioclase	31.7	30.2	30.9
	Amphibole/Pyroxene	7.51	7.15	6.87
	Chlorite	5.40	6.66	5.58
	Micas	7.58	8.52	9.90
	Clays	7.39	6.77	6.96
	Other Silicates	0.16	0.18	0.14
	Calcite	5.49	4.77	5.95
	Dolomite	16.3	16.1	13.8
	Apatite	0.69	0.72	0.66
	Other	0.13	0.18	0.11
	Total	100.0	100.0	100.0

Gold Deportment

A mineralogical gold deportment study was conducted on each sample and the procedure included pre-concentration by heavy liquid separation and superpanning. Optical microscopy and SEM-

EDS analysis methods were used for gold mineral scanning, identification, grain size measurement, and association characteristics. Chemical analyses were also used to determine the distribution of gold by grade and gold minerals by association category in the pre-concentration fractions, and weighted to the overall sample. A summary of the results are listed in Table 13.3.7.

Table 13.3.7 Mineralogical Characteristics of the Gold Minerals

Sample ID	Au Grade	Au Distribution by Association		Size Range (µm)	Average Size (µm)	Au-Mineral Abundance	Minerals Associated with Exposed and Locked Au-Minerals
High Grade Comp	**4.23**	Liberated	86.2	0.6 - 66.3	10.0	Gold (99%), Calaverite (1%)	Silc 48.2%, Py 42.0%, Qtz 4.76%, Bi-Te/Py 1.70%, Qtz/Silc, 1.21%, Py/Dol 1.08%, and other minerals <1%.
		Exposed		0.6 - 23.7	4.9		
		Locked	13.8	0.6 - 10.7	2.6		
			100	**0.6 - 66.3**	**5.9**		
Low Grade Comp	**0.83**	Liberated	82.4	0.6 - 73.7	12.9	Gold (100%)	Py 85.4%, Qtz 5.97%, Qtz/Ap/FeOx 3.62%, Silc/Py 2.08%, Silc 1.20 %, and other minerals <1%.
		Exposed		0.6 - 32.1	5.0		
		Locked	17.6	0.6 - 8.0	2.1		
			100	**0.6 - 73.7**	**4.4**		
Hanging Wall Comp	**1.85**	Liberated	73.7	0.8 - 78.0	10.1	Gold (97%), Calaverite (2%), Au-Te-Bi (1%)	Py 60.8%, Silc 19.5%, Dol 14.1%, Silc/Dol 4.36%, and other minerals <1%.
		Exposed		0.6 - 32.1	3.3		
		Locked	26.3	0.6 - 15.5	2.3		
			100	**0.6 - 78.0**	**3.3**		

In summary, the main gold mineral identified in all three composite samples was native gold (Au >75%, Ag <25%). Trace amounts of calaverite ($AuTe_2$), and unknown alloys of Au-Te-Bi were also identified. The grain size of the gold minerals was similar for all three domain composites. The average gold grain size for the HG, LG and HW domain composite samples was 5.9 µm, 4.4 µm, and 3.3 µm, respectively.

13.3.4 Comminution Testwork

A suite of comminution testwork was conducted to determine the variability of comminution parameters throughout the orebody and allow parameters for design of the comminution circuit to be derived.

Comminution tests conducted included the following:

- Unconfined Compressive Strength (UCS) determination.
- Bond Low-Energy Impact Crushing Work Index (CWi) determination.
- Bond Rod Mill Work Index (RWi) determination.
- Bond Ball Mill Work Index (BWi) determination.
- Bond Abrasion Index (Ai) determination.
- JK Drop Weight testwork.
- SMC testwork.

Most of the testwork was completed by SGS Lakefield, the results from the SMC tests were interpreted and ranked by JKTech and the UCS testwork was completed by SGS Beckley. The Bond rod mill Work index and Bond ball mill work index were also tested by Metso for comparative results. The SGS Lakefield results are listed in Table 13.3.8.

Table 13.3.8 Comminution Testwork Summary

Domain Composite Samples	Relative Density	DWi A x b[1]	SMC A x b[2]	SMC t_a[2]	Work Indices (kWh/t) CWi	Work Indices (kWh/t) RWi	Work Indices (kWh/t) BWi	Ai	UCS (MPa)
Deep HG	2.79	27.3	29.0	0.21	15.8	20.5	18.8	0.529	106.9
Shallow HG	2.80	27.2	28.0	0.29	14.0	20.7	16.5	0.382	106.6
Low Grade	2.74	28.4	26.7	0.33	14.0	19.6	16.9	0.273	38.1
Variability Samples	**Relative Density**	**DWi A x b[1]**	**SMC A x b[2]**	**SMC t_a[2]**	**Work Indices (kWh/t) CWi**	**Work Indices (kWh/t) RWi**	**Work Indices (kWh/t) BWi**	**Ai**	**UCS (MPa)**
Minimum	2.73	28.4	33.6	0.32	14.0	18.0	14.1	0.213	38.1
Maximum	2.85	27.2	26.0	0.24	15.8	21.5	19.7	0.879	106.9
Median	-	27.3	29.5	0.28	14.0	20.3	17.2	0.507	106.6
Average	2.78	-	-	-	-	-	-	0.525	-
85th Percentile	-	27.3	28.0	0.26	-	21.0	18.4	0.703	-
Design	2.78	27.3	28.1[3]	0.26	15.8	21.0	20.3[4]	0.703[5]	106.9

Note 1: Values from the DWT results

Note 2: Values from the SMC results

Note 3: Design Axb value derived from the 85th percentile ranking of specific energies determined for each individual ore type.

Note 4: Bond Ball Mill Work Index for Design includes a 10.0% correction factor to the 85th percentile value for different results of comparable samples at different laboratories.

Note 5: Abrasion Index Average value of 0.525 used for operating cost estimation and Design value of 0.703 used for wear type material selection.

Salient outcomes from the comminution testwork programme can be summarised as follows:

- Unconfined compressive strength (UCS) ranged from 38.1 MPa to 106.95 MPa. The UCS values indicate the ore is competent and can be primary crushed using a gyratory crusher.

- Crushing work indices (CWi) ranged from 14.0 to 15.8 kWh/t, with a median CWi of 14.0 kWh/t. The samples can be characterised as moderately hard to hard.

- Bond rod mill work indices (RWi) ranged from 18.0 to 21.5 kWh/t, with an 85th percentile RWi of 21.0 kWh/t. The comparative laboratory testing measured similar values for the domain composite samples.

- Bond ball mill work indices (BWi) ranged from 14.1 to 19.7 kWh/t, with an 85th percentile BWi of 18.4 kWh/t. The comparative laboratory testing results were higher for the domain composite samples. Therefore, the design value is 20.3 kWh/t, which includes a +10.0% correction factor to the 85th percentile value. The Bond rod and ball mill work indices are classified as hard to very hard indicating a high grinding energy requirement.

- The abrasion indices measured range from 0.213 to 0.879, with an average abrasion index of 0.525. The samples can be classified as medium to very abrasive. A review of the variability samples with very high abrasion index show that some Diorite is present in these particular samples. Diorite has an abrasion index range of 0.5 to 0.8 and could be the cause of some very high abrasion index results. An abrasion index average value of 0.525 is used for operating cost estimation and a design value of 0.703 is used for wear type material selection.

- The drop weight Index of the three domain composites tested showed similar results to the SMC Axb values measured and gives confidence in the results obtained. The Axb values measured range from 26.0 to 33.6 and ta values range from 0.32 to 0.24. The design Axb value of 28.1 was derived from the 85th percentile ranking of specific energies determined for each individual ore type. Overall, all of the samples are characterised as hard to very hard.

In general, the samples tested are classified as hard to very hard with medium to very abrasive properties. The ore is amenable to primary crushing followed by a SAG mill and ball mill with pebble crushing (SABC).

13.3.5 Gravity Concentration

Gravity concentration testwork was conducted on the domain composite sample to evaluate the recovery of gold. This work was completed by SGS Lakefield. The domain composites samples were also submitted for an Extended Gravity Recoverable Gold (E-GRG) testwork program, which was conducted by FLSmidth at Dawson Metallurgical Laboratories in Utah.

Table 13.3.9 Gravity Separation Summary by SGS Lakefield

Sample	Test No.	Feed Size P_{80}, µm	Product	Mass %	Assays, g/t Au	% Distribution Au
High Grade	G-3	76	Mozley Concentrate	0.26	347	**19.7**
			Knelson/Mozley Tailing	99.7	3.63	80.3
			Head (Calculated)	100.0	4.50	100.0
			Head (Direct)	100.0	4.23	
Low Grade	G-2	78	Mozley Concentrate	0.21	75.6	**19.0**
			Knelson/Mozley Tailing	99.8	0.68	81.0
			Head (Calculated)	100.0	0.84	100.0
			Head (Direct)	100.0	0.83	
Hanging Wall	G-1	73	Mozley Concentrate	0.30	168	**29.5**
			Knelson/Mozley Tailing	99.7	1.22	70.5
			Head (Calculated)	100.0	1.72	100.0
			Head (Direct)	100.0	1.85	

The testwork completed by SGS Lakefield shows the gravity separation gold recovery values ranged from 19.0% (LG) to 29.5% (HW). In general, the calculated head grades compared well to the direct head grades. These results indicated moderate gravity recoverable gold was present in the samples tested.

Table 13.3.10 Gravity Separation Summary by Dawson Met Lab

Composite	Head Grade (g/tonne)	Final Tails (g/tonne)	EGRG Value	Stage Recoveries			Final Grind Size (um)	AMIRA Classification
				1st	2nd	3rd		
Hanging Wall	1.60	0.95	40.9	11.5	11.0	18.4	68	Very Fine to Fine
High Grade	3.97	2.75	31.4	11.1	9.5	10.7	52	Very Fine to Fine
Low Grade	0.71	0.44	37.5	15.6	10.9	11.1	69	Very Fine to Fine

Further gravity testwork completed by Dawson Metallurgical Laboratories show the E-GRG recovery values after three stages on concentration ranged from 31.4% (HG) to 40.9% (HW). The results also showed that the concentrate gold grain sizes are within the very fine to fine classification.

The E-GRG test results were subsequently modelled. With about half of the gravity recoverable gold in the minus 25 micron size class, this gold size fraction will readily exit the grinding circuit in the cyclone overflow stream and will be challenging to recover at full scale. Modelling showed that a gravity circuit will have a marginal contribution at best. Based on these results, a gravity separation circuit is not warranted for the Fekola process flowsheet. All subsequent testwork has been completed without gravity separation.

13.3.6 Grind / Recovery Testwork

Grind and leach recovery tests were performed on the three metallurgical domain composites at grind sizes of 80% passing (P_{80}) ranging from 149 to 53 microns to determine the effect of grind size on gold recovery.

The cyanidation testwork was conducted in bottle rolls under the following conditions:

- Leach pulp density of 45% (w/w) solids in tap water.
- pH 10.5 to 11.0 adjusted with reagent grade hydrated lime.
- Initial cyanide dosage of 0.5 kg/t NaCN with residual sodium cyanide levels maintained at or above 500 ppm.
- Dissolved oxygen levels of 4 ppm or greater.
- 24 hour leach duration.

The grind / recovery leach results are summarised in Table 13.3.11.

Table 13.3.11 Summary of Grind / Recovery Leach Results of Whole Ore Cyanidation

Domain	CN Test No.	Feed Size P_{80}, µm	Reagent Cons. kg/t of CN Feed NaCN	CaO	% Au Extraction 2 h	4 h	8 h	**24 h**	Au Residue, g/t A	B	C	D	E	F	Avg.	Au Head, g/t Calc.	Direct	Avg. Calc.	**% Au Norm. Extr'n**
High Grade	15R	129	0.43	0.53	40	69	79	**84.1**	0.75	0.63	0.75	0.70	0.74	0.70	0.71	4.48	4.23	4.50	**84.2**
	16	100	0.44	0.59	39	67	80	**86.5**	0.59	0.57	0.69	0.60	0.63	0.64	0.62	4.59			**86.2**
	17	89	0.49	0.62	35	69	83	**87.7**	0.56	0.59	0.62	0.55	0.52	0.53	0.56	4.55			**87.6**
	18	78	0.73	0.65	25	73	85	**89.0**	0.49	0.47	0.53	0.49	0.46	0.47	0.49	4.41			**89.1**
	19	55	0.75	0.69	10	77	89	**92.0**	0.37	0.35	0.37	0.37	0.36	0.33	0.36	4.49			**92.0**
Low Grade	10	135	0.23	0.58	56	75	81	**84.6**	0.11	0.14	0.14	...	...	...	0.13	0.85	0.83	0.84	**84.6**
	11	102	0.32	0.57	51	76	85	**87.5**	0.10	0.10	0.11	...	...	...	0.10	0.82			**88.2**
	12	88	0.37	0.57	40	72	86	**89.4**	0.07	0.10	0.11	...	...	...	0.09	0.88			**89.3**
	13	70	0.41	0.59	55	62	69	**91.5**	0.07	0.07	0.07	...	...	...	0.07	0.82			**91.7**
	14	48	0.69	0.67	33	62	69	**92.1**	0.07	0.07	0.06	...	...	...	0.07	0.85			**91.7**
Hanging Wall	5	132	0.16	0.59	62	70	75	**87.3**	0.25	0.21	0.23	...	...	...	0.23	1.81	1.85	1.80	**87.3**
	6	97	0.34	0.57	59	72	77	**88.4**	0.23	0.21	0.19	...	...	...	0.21	1.81			**88.4**
	7	95	0.41	0.56	51	71	77	**90.2**	0.18	0.17	0.18	...	...	...	0.18	1.80			**90.0**
	8	73	0.46	0.57	47	72	80	**91.3**	0.14	0.16	0.16	...	...	...	0.15	1.76			**91.7**
	9	48	0.70	0.63	27	71	83	**93.8**	0.11	0.11	0.12	...	...	...	0.11	1.84			**93.9**

The results indicate:

- Lower residue grades and higher gold recoveries with increasing fineness of grind.
- The calculated gold head grades compared well to the direct head grade.
- Very little variation in the gold residue grades which indicated that the samples did not contain free gold or “nugget gold”.
- The leach kinetic results indicated that all of the samples continued to leach between 8 and 24 hours, however the conditions had not been optimised. The leach retention time will be tested during the leach conditions optimisation testwork program.
- A grind optimisation study was updated with the results listed in Table 13.3.11 to evaluate the effect of grind size on project economics. The evaluation compared gold revenue against operating and capital expenditure for the grind sizes considered. A grind size (P_{80}) of 75 microns is considered to be the economic optimum for the Project.

13.3.7 Preg Robbing Assessment

An evaluation of the preg robbing potential of the three metallurgical domain composites was conducted under the following conditions:

- P_{80} grind of 75 microns.
- Leach pulp density of 45% (w/w) solids in tap water.
- pH 10.5 to 11.0 adjusted with reagent grade hydrated lime.

- Initial cyanide dosage of 0.5 kg/t NaCN with residual sodium cyanide levels maintained at or above 500 ppm.
- Dissolved oxygen levels of 4 ppm or greater.
- 24 hour leach duration.
- With and without activated carbon.

Four domain composite samples were tested. One sample was tested using a direct leach and the comparative sample was tested in the presence of activated carbon, 10 g/L. Results from testwork are presented in Table 13.3.12.

Table 13.3.12 Summary of Whole Ore vs CIL Cyanidation

Domain	CN Test No.	Whole Ore or CIL	Feed Size P_{80}, µm	Reagent Cons. kg/t of CN Feed NaCN	CaO	% Au Extraction 2 h	4 h	8 h	**24 h**	Au Residue, g/t A	B	C	D	E	F	**Avg.**	Au Head, g/t Calc.	Direct
High Grade	18	W.O.	78	0.73	0.65	25	73	85	**89.0**	0.49	0.47	0.53	0.49	0.46	0.47	**0.49**	4.41	4.23
	22	CIL	69	0.67	0.62	...	...	...	**90.2**	0.43	0.41	0.44	...	...	...	**0.43**	4.36	
	23	W.O.	73	0.36	0.91	78	84	88	**89.7**	0.43	0.44	0.44	...	...	...	**0.44**	4.23	
	24	CIL	73	0.32	1.00	...	...	...	**89.4**	0.47	0.45	0.42	...	...	...	**0.45**	4.22	
Low Grade	13	W.O.	70	0.41	0.59	55	62	69	**91.5**	0.07	0.07	0.07	...	...	...	**0.07**	0.82	0.83
	21	CIL	64	0.62	0.71	...	...	...	**90.7**	0.07	0.08	0.07	...	...	...	**0.07**	0.79	
Hanging Wall	8	W.O.	73	0.46	0.57	47	72	80	**91.3**	0.14	0.16	0.16	...	...	...	**0.15**	1.76	1.85
	20	CIL	66	0.68	0.69	...	...	...	**92.0**	0.15	0.14	0.15	...	...	...	**0.15**	1.84	

The results indicate:

- No preg robbing characteristics.

The testwork results show the samples did not illustrate any preg-robbing characteristics even though the feed grind size (P_{80}) did vary slightly for the comparative tests. Additional tests were conducted for the High Grade domain composite because the initial feed size varied by 9 microns. A repeat test confirmed that preg-robbing was not occurring for this sample and the residue grades were essentially identical.

13.3.8 Whole Ore Leach Cyanidation Optimisation Testwork

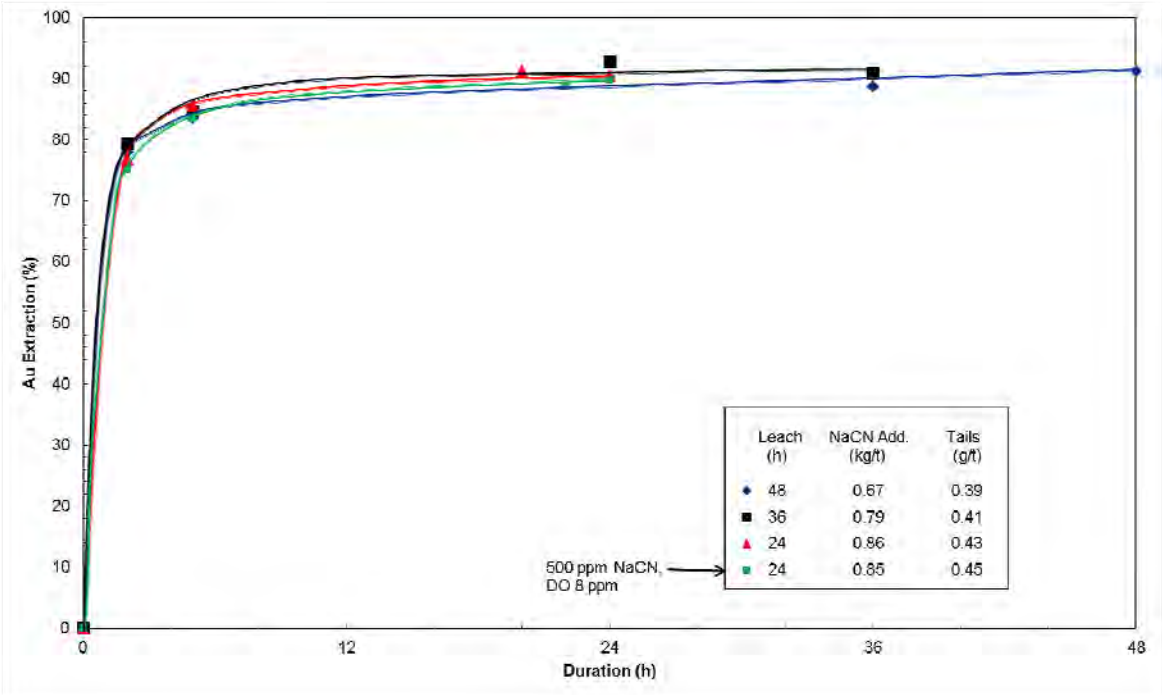
The main cyanide consumer in the Fekola deposit is pyrite. Pyrite is an iron sulphide mineral that has a varying detrimental effect on the cyanidation of gold. The detrimental effect is related to the mineralogical form of pyrite and the dissolution of iron and sulphide that precipitate on gold grains; decreasing gold leaching kinetics. This detrimental effect can be alleviated by the addition of lead nitrate and oxygen.

Residence Time Determination

Testwork to identify when gold dissolution reaches steady state was completed with the following conditions:

- P_{80} grind of 75 microns.
- Leach pulp density of 45% (w/w) solids in distilled water.
- Leach pulp temperature of 30°C.
- pH 10.3 to 10.5 adjusted with reagent grade hydrated lime.
- Initial cyanide dosage of 0.35 kg/t NaCN with residual sodium cyanide levels maintained at or above 350 ppm.
- Dissolved oxygen levels of 14 to 17 ppm.
- 24, 36, and 48 hour leach duration.

Figure 13.3.2 Effect of Residence Time



Leaching profiles in Figure 13.3.2 show very fast gold dissolution with a steady state almost reached in 24 hours. The overall extraction was 90.5% (leach residue at 0.43 g/t Au). This extraction is 0.7% lower than the overall extraction obtained in 48 hours. It was considered that a maximum extraction of gold would be reached in 24 hr with further improvement of leaching conditions.

It was not possible to determine the effect of retention time on cyanide addition because most of the adjustments of cyanide concentration were performed in the first 6 hours. It is interesting to point out that the tests with 350 ppm NaCN and DO of ~15 ppm showed leaching kinetics almost identical to leaching kinetics obtained with 500 ppm NaCN and DO of 8 ppm (green plot in Figure 13.3.2).

Lead Nitrate Addition

Testwork to identify optimum lead nitrate addition was completed with the following conditions:

- P_{80} grind of 75 microns.
- Leach pulp density of 45% (w/w) solids in distilled water.
- Leach pulp temperature of 30°C.
- pH 10.3 – 10.5 adjusted with reagent grade hydrated lime.
- Initial cyanide dosage of 0.35 kg/t NaCN with residual sodium cyanide levels maintained at or above 350 ppm.
- Dissolved oxygen levels of 14 to 17 ppm.
- 24 hr leach duration.
- Initial addition of lead nitrate at 0, 50, 100 and 250 g/t of $Pb(NO_3)_2$.

Figure 13.3.3 Effect of Lead Nitrate Addition

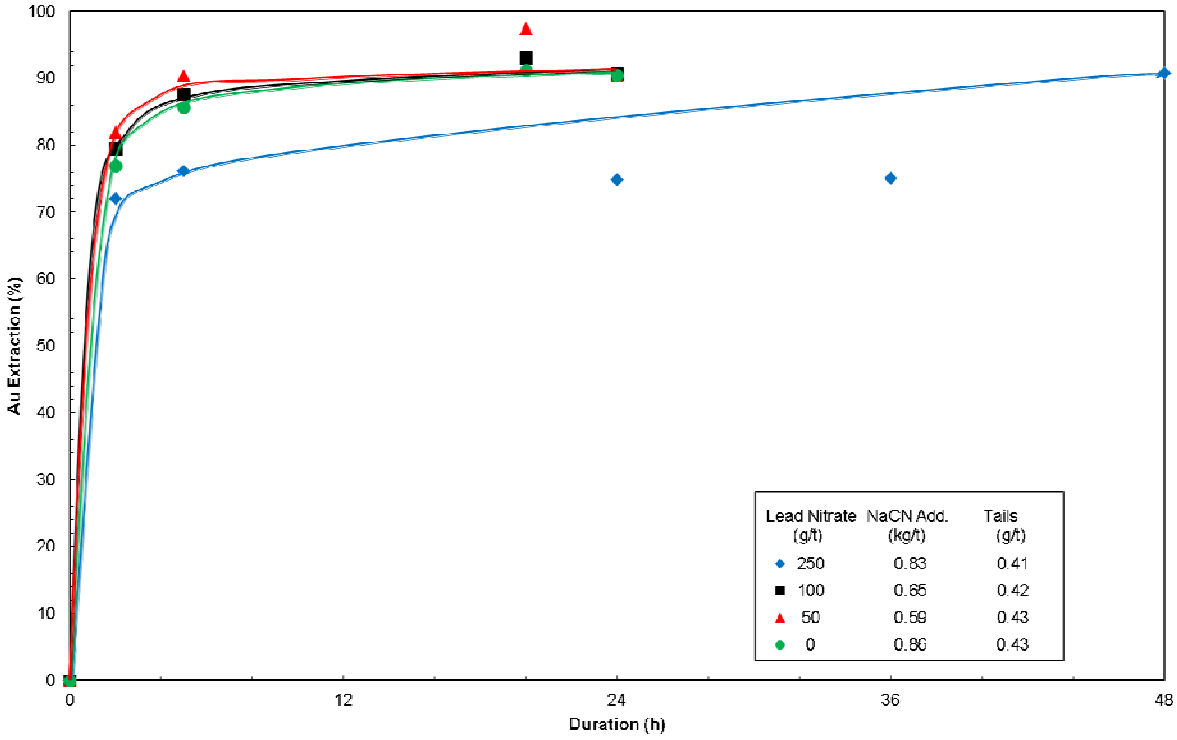


Figure 13.3.3 illustrates the effect of lead nitrate added at the beginning of cyanidation. The concentration of cyanide was 350 ppm NaCN and the DO was 14 to 17 ppm. Test 6 from this series, with elevated DO and 100 g/t lead nitrate addition, produced the highest gold extraction. There was basically no effect of lead nitrate addition on gold extraction at 50 g/t and lead nitrate became detrimental to the gold leaching rate when added at 250 g/t. Lead nitrate addition is considered beneficial when processing ore zones with varying and higher sulphide content. Consequently, the testwork program was continued with the addition of 100 g/t lead nitrate.

Cyanide Concentration Determination

Testwork to identify optimum cyanide concentration determination was completed with the following conditions:

- P_{80} grind of 75 microns.
- Leach pulp density of 45% (w/w) solids in distilled water.
- Leach pulp temperature of 30°C.
- pH 10.3 – 10.5 adjusted with reagent grade hydrated lime.
- Dissolved oxygen levels of 14 to 17 ppm.
- 24 hr leach duration.

- Initial addition of lead nitrate at 100 g/t of $Pb(NO_3)_2$.
- Initial cyanide dosage ranging from 0.15 to 0.55 kg/t NaCN with residual sodium cyanide levels maintained at or above dosage level.

Figure 13.3.4 Effect of Cyanide Concentration

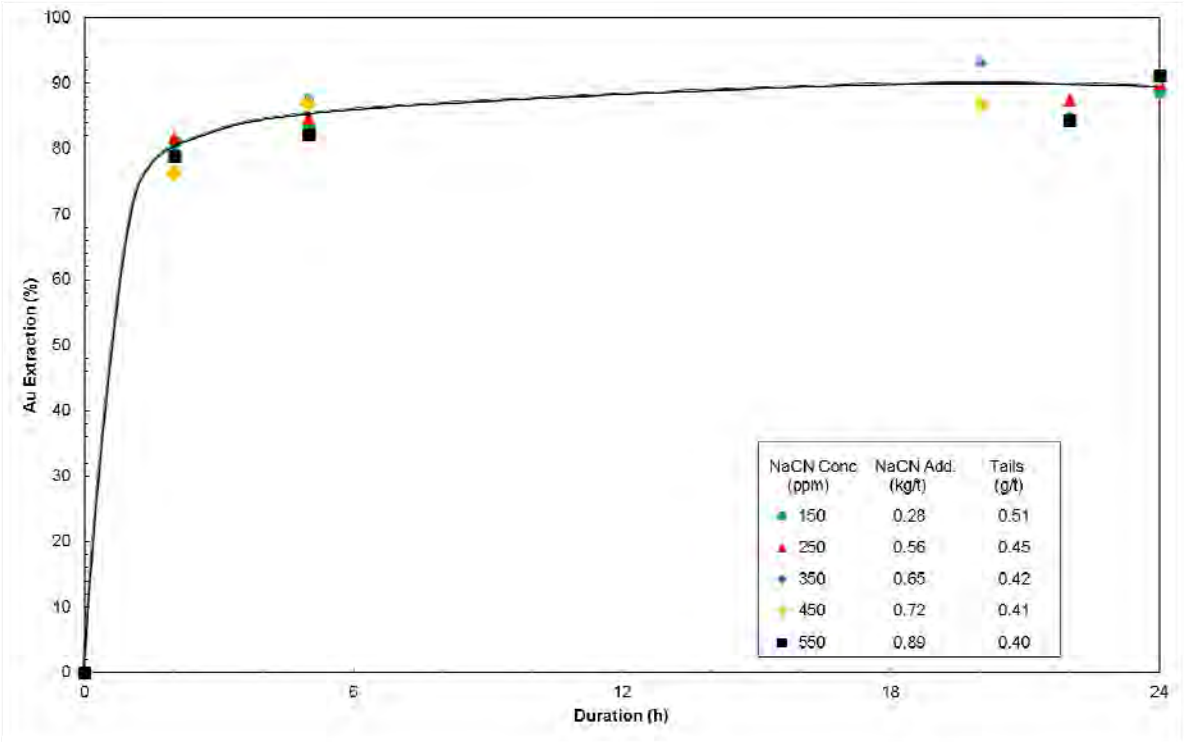


Figure 13.3.4 shows the effect of cyanide concentration ranging from 150 ppm to 550 ppm NaCN. There was no significant difference in the leaching kinetics in the range of concentrations tested. The gold content of the leach residues increased when the cyanide concentration was reduced to 250 ppm and lower. This corresponds to a decrease of gold extraction from 91.1% to 90.0%. The cyanide addition decreased from 0.89 kg/t NaCN to 0.65 kg/t when the cyanide concentration decreased from 550 ppm NaCN to 350 ppm NaCN respectively.

Dissolved Oxygen Level Determination

Testwork to identify optimum dissolved oxygen level determination was completed with the following conditions:

- P_{80} grind of 75 microns.
- Leach pulp density of 45% (w/w) solids in distilled water.
- Leach pulp temperature of 30°C.
- pH 10.3 to 10.5 adjusted with reagent grade hydrated lime.

- 24 hr leach duration.
- Initial addition of lead nitrate at 100 g/t of $Pb(NO_3)_2$.
- Initial cyanide dosage of 0.35 kg/t NaCN with residual sodium cyanide levels maintained at or above 350 ppm.
- Dissolved oxygen levels of 8 and 15 ppm.

Figure 13.3.5 Effect of Dissolved Oxygen

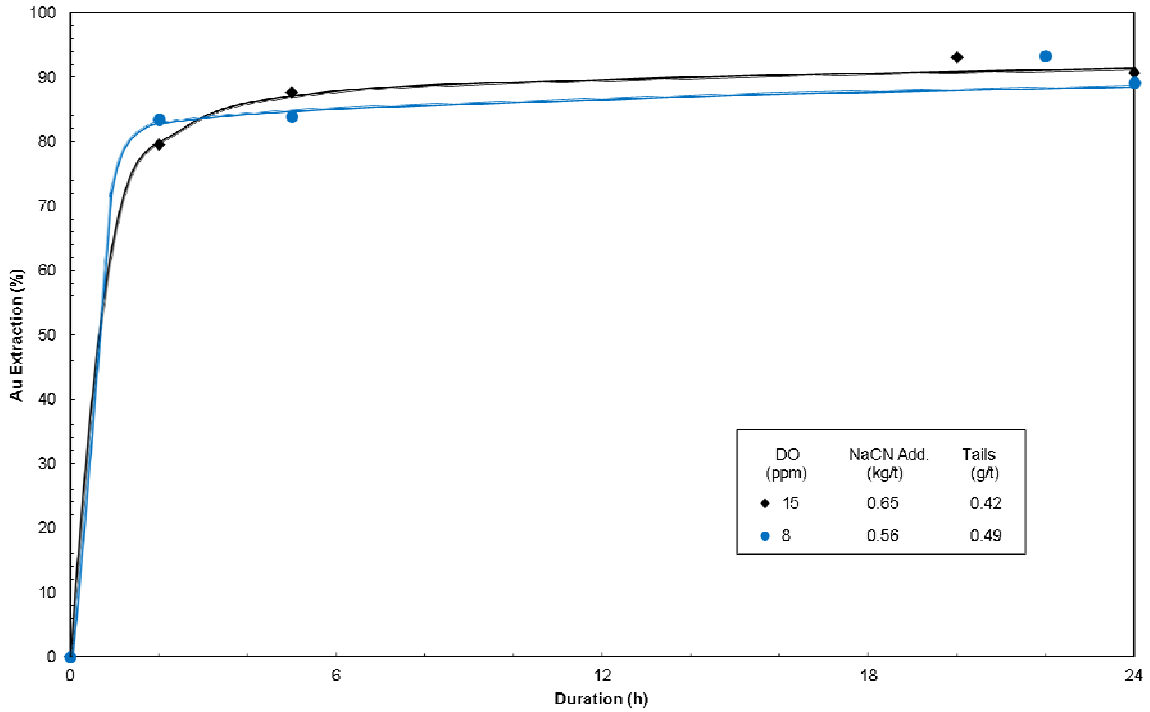


Figure 13.3.5 shows a decrease of dissolved oxygen concentration from 15 ppm to 8 ppm resulted in a reduction of gold extraction from 90.7% to 89.1%. This shows the importance of high dissolved oxygen to optimise gold extraction.

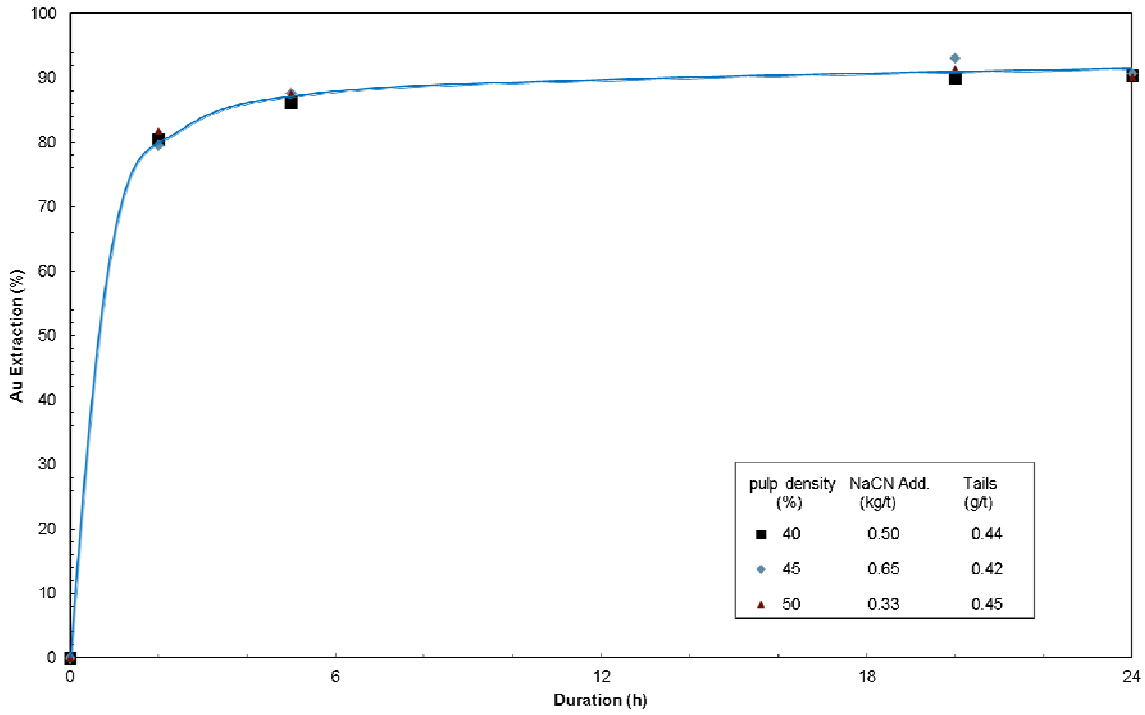
Pulp Density Determination

Testwork to identify optimum pulp density was completed with the following conditions:

- P_{80} grind of 75 microns.
- Leach pulp temperature of 30°C.
- pH 10.3 to 10.5 adjusted with reagent grade hydrated lime.
- 24 hr leach duration.

- Initial addition of lead nitrate at 100 g/t of $Pb(NO_3)_2$.
- Initial cyanide dosage of 0.35 kg/t NaCN with residual sodium cyanide levels maintained at or above 350 ppm.
- Dissolved oxygen levels of 14 to 17 ppm.
- Leach pulp density of 40%, 45% and 50% (w/w) solids in distilled water.

Figure 13.3.6 Effect of Pulp Density



According to Figure 13.3.6, the variation of pulp density from 40 to 50% showed no significant difference in terms of leaching kinetics; however, the overall gold extractions were slightly lower compared to 45% solids. Therefore, 45% solids were selected as the optimum leach pulp density.

Reproducibility of Optimum Gold Leaching Conditions

Testwork to confirm reproducibility of optimum gold leaching conditions for the Fekola deposit was completed with the following conditions:

- P_{80} grind of 75 microns.
- Leach pulp density of 45% (w/w) solids in distilled water.
- Leach pulp temperature of 30°C.

- pH 10.3 to 10.5 adjusted with reagent grade hydrated lime.
- 24 hr leach duration.
- Initial addition of lead nitrate at 100 g/t of $Pb(NO_3)_2$.
- Initial cyanide dosage of 0.35 kg/t NaCN with residual sodium cyanide levels maintained at or above 350 ppm.
- Dissolved oxygen levels of 14 to 17 ppm.

Figure 13.3.7 Reproducibility of Optimum Gold Leaching Conditions

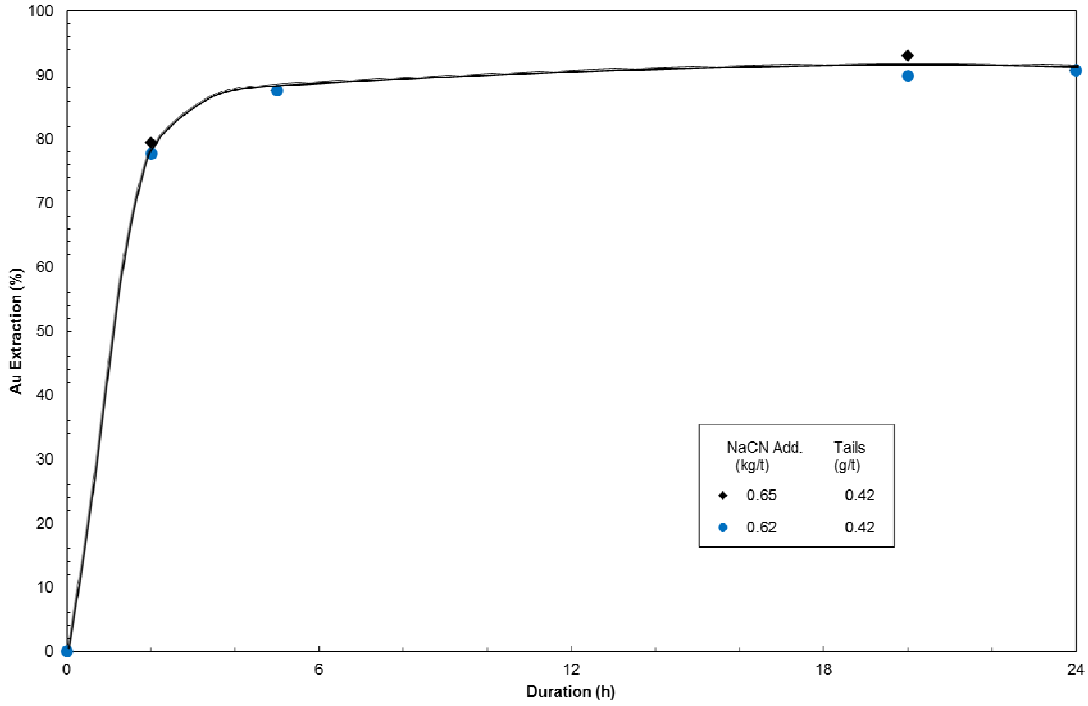


Figure 13.3.7 shows reproducibility of optimum gold leaching conditions. The optimised conditions are 24 hour leach retention time, 14 to 17 ppm DO concentration, 350 ppm NaCN concentration, 100 g/t lead nitrate addition and 45% solids. The leaching profiles and gold content of leach residues indicates that the optimum conditions are highly reproducible.

Summary of Optimum Gold Leaching Conditions

The Fekola high grade domain composite sample used for the gold leaching optimisation testwork programme contained 4.23 g/t Au. The mineralogical composition of the sample was mostly gangue minerals with 3.45% pyrite. Leaching profiles showed that most of the gold dissolved within 12 hour and unleached gold is about 9 to 10%. The optimum leaching conditions identified are 24 hour cyanidation with 350 ppm NaCN, 100 g/t lead nitrate, pH 10.3 to 10.5, DO ~15 ppm and a pulp density of 45% solids (w/w). The gold extraction was 91% (leach residue of 0.42 g/t) and cyanide addition was 0.65 kg/t NaCN. Gold extraction is influenced by the oxygen content of

the slurry. Gold extraction was 2% lower when the DO was reduced from 15 ppm to 8 ppm. At a cyanide concentration of 250 ppm NaCN, gold extraction was reduced by 1.0%. Increased addition of lead nitrate to 250 g/t significantly reduced the leaching kinetics. The optimum gold leaching conditions were found to be highly reproducible.

13.3.9 Whole Ore Cyanidation of Variability Samples at Optimised Leach Conditions

Testwork on the metallurgical variability samples was completed with the optimum gold leaching conditions identified in the testwork programme described in Section 13.3.8. The optimum gold leaching conditions are:

- P_{80} grind of 75 microns.
- Leach pulp density of 45% (w/w) solids.
- Leach pulp temperature of 30°C.
- pH 10.3 to 10.5 adjusted with reagent grade hydrated lime.
- 24 hr leach duration.
- Initial addition of lead nitrate at 100 g/t of $Pb(NO_3)_2$.
- Initial cyanide dosage of 0.35 kg/t NaCN with residual sodium cyanide levels maintained at or above 350 ppm.
- Dissolved oxygen levels of 14 to 17 ppm.
- Table 13.3.13 lists a summary of these results. The testwork included seven HG, seven LG and four HW metallurgical variability samples. The samples with a gold content >5.0 g/t were repeated with an initial cyanide dosage of 0.50 kg/t NaCN with residual sodium cyanide levels maintained at or above 500 ppm.

Table 13.3.13 Summary of Whole Ore Cyanidation of Variability Samples at Optimised Leach Conditions

Domain	Sample	CN Test No.	Feed Size P_{80}, µm	Reagent Addition Kg/t on CN Feed		% Au Extraction 24 h	Au Residue, g/t			Au Head, g/t		
				NaCN	CaO		A	B	C	Avg.	Calc	Direct
High Grade	FEK-MET-004	35	72	0.82	0.68	**96.9**	0.32	0.31	0.33	0.32	10.2	11.2
	*FEK-MET-004**	*56*	*79*	*1.03*	*0.66*	***97.7***	*0.24*	*0.25*	*0.26*	*0.25*	*10.8*	*11.2*
	FEK-MET-005	36	75	0.88	0.62	**95.8**	0.46	0.50	0.49	0.48	11.6	12.7
	*FEK-MET-005**	*57*	*73*	*1.09*	*0.61*	***96.0***	*0.45*	*0.48*	*0.47*	*0.47*	*11.5*	*12.7*
	FEK-MET-006	37	74	0.81	0.64	**95.8**	0.19	0.18	0.21	0.19	4.58	4.12
	FEK-MET-007	38	67	0.81	0.58	**94.6**	0.23	0.23	0.24	0.23	4.35	4.65
	FEK-MET-008	39	79	0.92	0.75	**87.3**	0.39	0.48	0.48	0.45	3.53	3.56
	FEK-MET-009	40	70	0.83	0.56	**94.5**	0.30	0.24	0.26	0.27	4.84	5.88
	*FEK-MET-009**	*58*	*72*	*1.09*	*0.59*	***94.8***	*0.25*	*0.26*	*0.25*	*0.25*	*4.84*	*5.88*
	FEK-MET-010	41	70	0.78	0.75	**94.3**	0.14	0.16	0.15	0.15	2.64	2.78
	Domain Comp	53	76	0.81	0.69	**90.2**	0.43	0.44	0.43	0.43	4.44	4.23
	Average	**...**	**72**	**0.84**	**0.65**	**94.2**	**...**	**...**	**...**	**0.30**	**5.96**	**6.41**
	Min	...	67	0.78	0.56	87.3	...	...	...	0.15	2.64	2.78
	Max	...	79	0.92	0.75	96.9	...	...	...	0.48	11.6	12.7
Low Grade	FEK-MET-015	46	68	0.62	0.83	**90.5**	0.05	0.05	0.06	0.05	0.56	0.58
	FEK-MET-016	47	71	0.89	0.58	**91.8**	0.06	0.08	0.07	0.07	0.86	0.98
	FEK-MET-017	48	71	0.78	0.76	**92.4**	0.09	0.10	0.10	0.10	1.28	1.49
	FEK-MET-018	49	69	0.71	0.68	**94.9**	0.06	0.06	0.06	0.06	1.17	1.08
	FEK-MET-019	50	75	0.76	0.60	**91.8**	0.06	0.07	0.06	0.06	0.77	0.75
	FEK-MET-020	51	72	0.82	0.65	**91.3**	0.07	0.08	0.07	0.07	0.84	0.84
	FEK-MET-021	52	67	0.87	0.62	**91.9**	0.07	0.08	0.08	0.08	0.95	0.92
	Domain Comp	55	72	0.72	0.66	**90.4**	0.08	0.08	0.08	0.08	0.83	0.83
	Average	**...**	**70**	**0.78**	**0.67**	**92.1**	**...**	**...**	**...**	**0.07**	**0.92**	**0.95**
	Min	...	67	0.62	0.58	90.5	...	...	...	0.05	0.56	0.58
	Max	...	75	0.89	0.83	94.9	...	...	...	0.10	1.28	1.49

Domain	Sample	CN Test No.	Feed Size P_{80}, µm	Reagent Addition Kg/t on CN Feed		% Au Extraction 24 h	Au Residue, g/t			Au Head, g/t		
				NaCN	CaO		A	B	C	Avg.	Calc	Direct
Hanging Wall	FEK-MET-011	42	71	0.82	0.69	**92.5**	0.13	0.14	0.12	0.13	1.73	1.68
	FEK-MET-012	43	74	0.68	0.77	**93.6**	0.08	0.08	0.07	0.08	1.20	1.29
	FEK-MET-013	44	68	0.73	0.58	**93.9**	0.13	0.14	0.15	0.14	2.29	2.67
	FEK-MET-014	45	75	0.77	0.63	**92.4**	0.16	0.19	0.18	0.18	2.33	2.33
	Domain Comp	54	74	0.78	0.63	**91.2**	0.15	0.15	0.16	0.15	1.75	1.85
	Average	**...**	**72**	**0.75**	**0.67**	**93.1**	**...**	**...**	**...**	**0.13**	**1.89**	**1.99**
	Min	...	68	0.68	0.58	92.4	...	...	...	0.08	1.20	1.29

***Repeat tests on select samples (>5 g/t Au) – using 0.5 g/L NaCN. These test results are not included in the Average, Min or Max values shown.**

The gold extractions for High Grade domain variability samples ranged from 87.3 to 96.9%, with an average extraction of 94.2% (seven samples). The FEK-MET-008 variability sample was the only sample which had a gold extraction <90%. The other six variability samples had gold extractions that ranged from 94.3 to 96.9%. The High Grade domain composite sample gold extraction was 90.2%, lower than the majority of the variability results. The three repeat cyanidation tests (*HG variability samples with heads grades >5 g/t*) which maintained a higher cyanide concentration (0.5 g/L NaCN) had slightly higher gold extractions when compared to the tests completed at the optimum conditions. The gold extractions were 0.2 to 0.8% higher.

The Low Grade domain variability gold extractions ranged from 90.5 to 94.9%, with an average of 92.1% (seven samples). The Low Grade domain composite sample gold extraction was 90.4%, slightly below the variability sample average.

The Hanging Wall domain variability gold extractions ranged from 92.4 to 93.9%, with an average of 93.1% (four samples). The Hanging Wall domain composite sample gold extraction was 92.7%. The average cyanide and lime consumptions for the HW domain composite tests were the lowest of the three domains.

In general, the calculated gold head grades compared well to the direct heads. The largest variations were in the high grade variability sample test results. The relationship between gold extraction and head grade for the variability samples is illustrated in Figure 13.5.1.

13.3.10 Diagnostic Leaching Testwork

Two diagnostic leach tests were conducted on High Grade cyanidation leach residue samples to determine the gold association and determine if the remaining gold was cyanide leachable. The residues selected for diagnostic leach testing were from test CN-39, a HG variability sample, FEK-MET-008, and from test CN-53, the High Grade domain composite sample. The residues were selected due to their higher gold content (0.45 g/t Au for CN-39 and 0.43 g/t Au for CN-53) and lower gold leach extractions (87.3% for test CN-39 and 90.2% for test CN-53).

The diagnostic leach results show:

- The overall gold extraction only increased slightly (1.4% for FEK-MET-008 and 0.6% for HG domain composite) after intensive cyanidation for 24 hrs.
- Regrinding the residues followed by additional intensive cyanidation leaching for another 24 hr improved the overall gold extraction significantly for both samples (6.7% for FEK-MET-008 and 6.8% for the HG Domain Composite). The increase in gold extractions with fine grinding indicated improved liberation and accessibility of exposed gold to the cyanide leach solution.
- Finally, the leach residues were subjected to hot aqua regia digestion to extract the gold associated with sulphide minerals. Gold extractions of 3.5% (FEK-MET-008) and 2.2% (HG domain composite) were achieved.

- The remaining gold, 1.1% in FEK-MET-008 and 0.2% in the HG domain composite, is considered to be as gold locked in silicates or gold associated with fine sulphides locked in silicates. It should be noted that the final aqua regia residues had sulphide sulphur grades of 0.07% and <0.05%, which indicated that essentially all of the sulphides present in the samples were dissolved.

Review of the origin of these samples does not reveal any apparent reason for the lower gold extraction rates or the diagnostic leach results observed. The fact that the leach residues samples needed regrinding before intensive cyanidation improved gold extraction could be considered to be related to the presence of trace amounts of Calaverite ($AuTe_2$), which is a slow leaching mineral that benefits from fine grinding. However the tellurium content of the high grade domain composite was below the detection limit and only the low grade domain composite had measurable tellurium in the head analysis. The variability samples were not analysed for tellurium. In summary, the most plausible explanation for the lower gold extractions is the incomplete liberation and exposure of finely disseminated gold particles based on the diagnostic leach results.

13.3.11 Bulk Cyanidation Tests

Three bulk cyanidation tests were completed to create samples for downstream testwork, which included cyanide destruction (CND), thickener testing, rheology and environmental testwork. The bulk tests were completed using the High Grade domain composite and the optimised leach conditions were applied. Upon completion of the cyanidation tests, the leached pulp was contacted with carbon for six hours to simulate a CIP circuit, prior to CND testing. Table 13.3.14 summarises the bulk testwork results.

Table 13.3.14 Summary of Bulk Whole Ore Cyanidation Tests

CN Test No.	Purpose	Sample Size kg	Feed Size P_{80}, µm	Reagent Cons. kg/t of CN Feed NaCN	Reagent Cons. kg/t of CN Feed CaO	**% Au Extraction 24 h**	Au Residue, g/t A	Au Residue, g/t B	Au Residue, g/t Avg.	Au Head, g/t Calc.	Au Head, g/t Direct
CN-63	CND Design	10	81	0.30	0.75	87.8	0.56	0.61	0.59	4.81	
CN-64	Thickener, Rheology, CND/Enviro.	40	71	0.68	0.78	89.5	0.52	0.45	0.49	4.61	4.23
CN-64R	CND/Envrio.	20	77	0.47	0.74	85.6	0.61	0.69	0.65	4.51	

The bulk cyanidation gold extractions were slightly lower than the 1 kg bottle roll test results (90.2%). The gold extractions ranged from ~86 to 90%. The optimum gold leaching conditions were found to be moderately reproducible with the bulk leach of the HG domain composite.

13.3.12 Cyanide Destruction Testwork

Three cyanide destruction tests were conducted on bulk High Grade domain composite leached pulp. The objective of the testwork was to obtain optimised conditions for the cyanide destruction circuit and produce treated pulp containing <10 mg/L residual CN_{WAD} using the SO_2 / air destruction process.

The main parameters adjusted in the testwork were the stoichiometric addition of sodium metabisulphite and the addition of copper. The tests were completed in batch and continuous mode. The Fekola pulp responded well to cyanide destruction treatment using the SO_2/air process. All of the tests were conducted at 30°C, 45% solids (w/w), pH 8.5, and a retention time of 60 minutes.

Upon completion of the CND program, the following operating conditions were recommended for the Fekola tailings pulp:

- Addition ratio of ~4 g of SO_2 per gram of gram of $CN_{WAD.}$
- Copper addition of 10 mg/L.
- pH = 8.5.
- 90 minutes residence time.

Testwork completed indicates the SO_2 / air process can be successfully employed to treat the CIP tailings stream to reduce CN_{WAD} levels.

13.3.13 Oxygen Uptake Testwork

Oxygen uptake tests were completed on the three domain composites, HG, LG and HW, at optimised leach conditions. The rate at which oxygen was consumed was determined over a 24 hour period. The oxygen uptake rate (mg/L/min) was calculated at 1, 2, 4, 6, and 24 hours. The initial tests completed only sparged oxygen into the pulp for 45 minutes prior to a 15 minute reading when the oxygen decay was measured. After the initial tests, duplicate tests (R1 and R2) were completed where the oxygen concentration was maintained at the design value (13 to 15 ppm) during the tests. For the triplicate and average test results, the relationship between oxygen uptake and time is illustrated in Figures 13.3.8 to 13.3.10.

Figure 13.3.8 Oxygen Demand of High Grade Domain Composite Sample

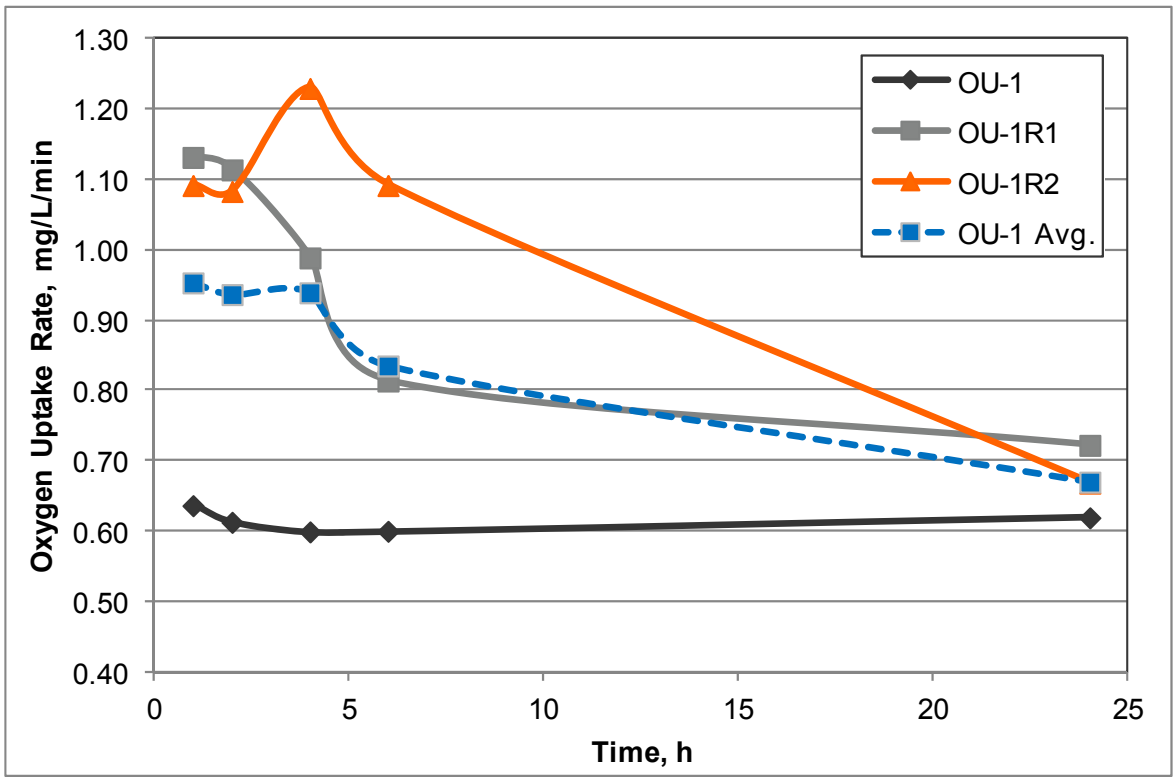


Figure 13.3.9 Oxygen Demand of Low Grade Domain Composite Sample

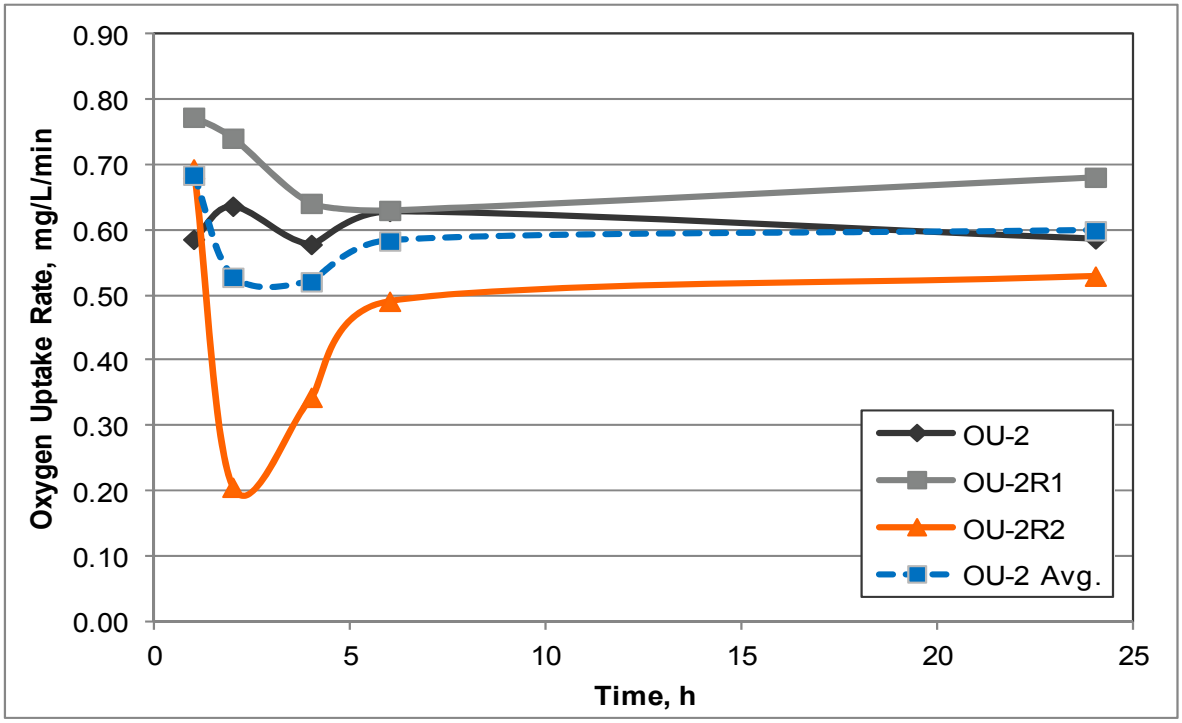
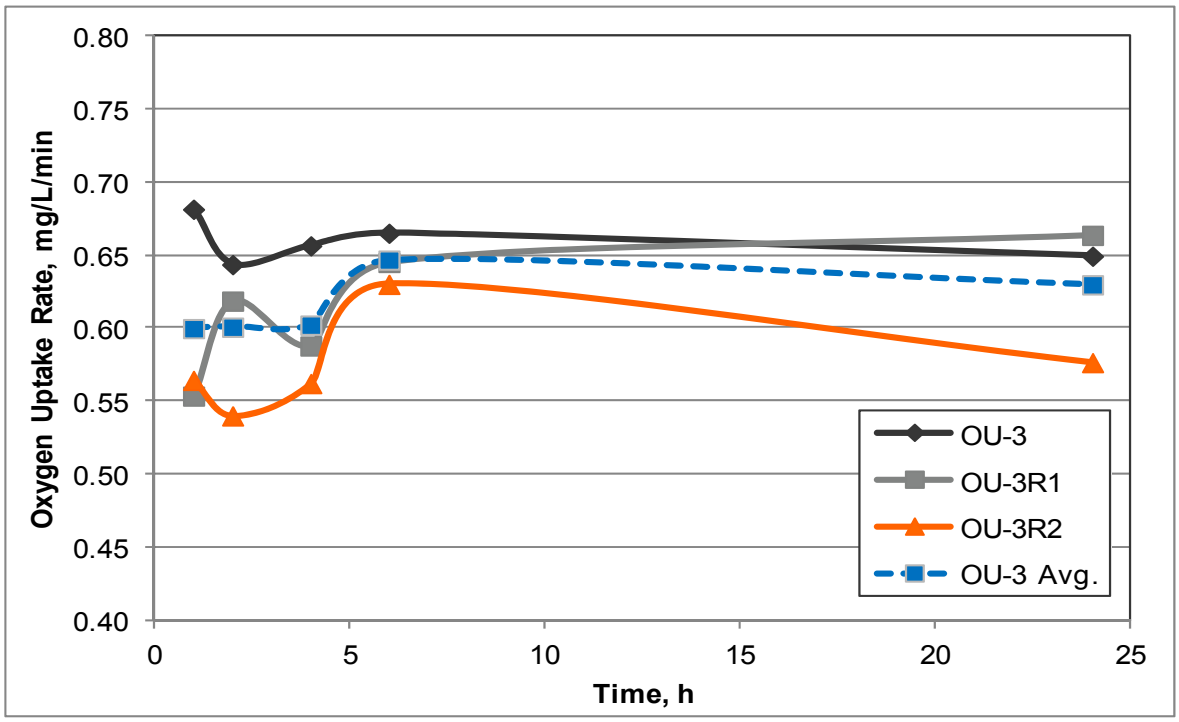


Figure 13.3.10 Oxygen Demand of Hanging Wall Domain Composite Sample



The results indicate the following:

- All three domain composite samples exhibit high oxygen demand for the duration of the test period.

The addition of oxygen, instead of air, is required to achieve the required dissolved oxygen levels of 13 to 17 ppm in the leach circuit.

13.3.14 Carbon Modelling

A high grade domain composite sample was leached in bulk at the optimised leach conditions for subsequent carbon adsorption and equilibrium testwork described below.

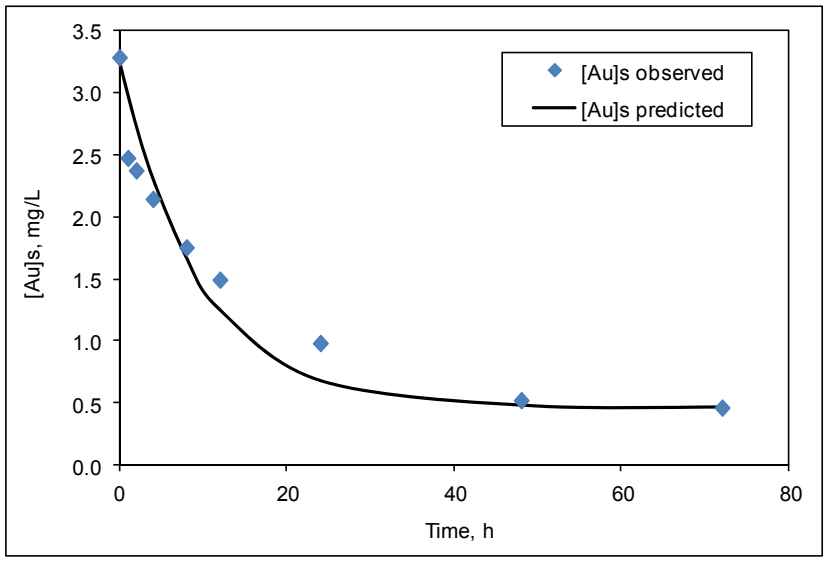
Carbon Adsorption Kinetics

The bulk leached pulp was then contacted with regenerated plant carbon in a batch test in a rolling bottle for 72 hours. Solution samples were removed for gold analysis at specific time intervals (0, 1, 2, 4, 8, 12, 24, 48, and 72 hrs) to establish the adsorption kinetic profile.

Table 13.3.15 Carbon Adsorption of High Grade Domain Composite Sample

Time h	Solution, mg/L Au	Calculated Loading, g/t Au
0	3.28	0
1	2.47	2401
2	2.37	2695
4	2.14	3364
8	1.75	4487
12	1.49	5229
24	0.98	6675
48	0.52	7959
72	0.46	8125

Figure 13.3.11 Carbon Adsorption of Bulk Leached HG Composite Sample

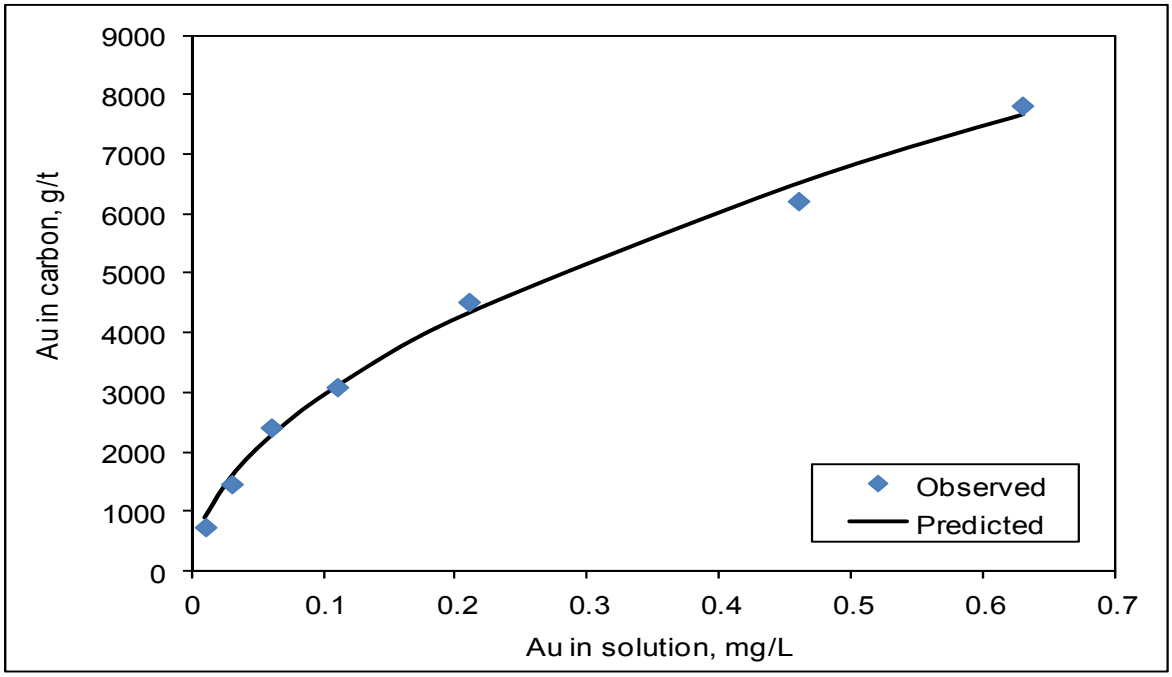


The kinetic (k) and equilibrium (K) adsorption constants generated from the sample tested were 0.015 h^{-1} and 17,731 g/t, respectively. The kinetic constant k is a first order rate constant that describes the rate of film diffusion of gold cyanide onto carbon in the initial stages of the adsorption process, while the equilibrium constant K represents the predicted equilibrium loading of gold cyanide on the carbon (in g/t) from a solution containing 1 mg/L Au. The product of the two constants (kK) provides a useful guide as to whether or not CIP/CIL will perform efficiently when processing the particular pulp. In this regard, experience has shown that when the kK value is <50 h^{-1} the carbon adsorption process will be slow and inefficient (requiring many adsorption stages and high carbon inventories). When the kK value is >100 h^{-1}, however, good CIP / CIL performance can be expected. Against this background, it can be seen that the Fekola HG Domain Comp gold adsorption properties are very good, with a value 260 h^{-1}. The adsorption kinetics (k = 0.015) are relatively slow but the equilibrium loading (K = 17,731) is favourable for this sample.

Carbon Adsorption Equilibrium Isotherms

Equilibrium isotherms for gold cyanide loading on activated carbon can generally be fitted well to a Freundlich isotherm via a non-linear regression. To generate the equilibrium data, predetermined amounts of pulp from the bulk leach test were contacted with predetermined amounts of activated carbon to achieve target solution/carbon ratios (or targeted carbon loadings). The slurries were agitated, in rolling bottles with carbon, for 72 hrs to "equilibrate" the activated carbon with the gold in solution, after which time the carbon was recovered by screening and samples of the slurry were filtered to generate carbon, solution and washed residue samples for gold and silver analysis. The results from the isotherm tests are plotted in Figure 13.3.12.

Figure 13.3.12 Equilibrium Loading of Gold onto Carbon from Bulk Leached HG Composite Sample



The isotherm shown in Figure 13.3.12 does not represent true equilibrium between gold in solution and on the carbon, but is rather considered to be a pseudo-isotherm that is representative of a practical loading limit in an operating plant. In practice, gold cyanide loading on activated carbon is so slow when it is close to equilibrium that the attainment of true equilibrium takes weeks or even months to be reached. But these much higher loading values are of little relevance as far as plant operations are concerned.

Carbon Adsorption Testing

The results of the carbon adsorption testwork are displayed in Figure 13.3.11 and Figure 13.3.12. The carbon adsorption constants have been calculated to be:

Kinetic adsorption constant **k**, h^{-1}	0.015

Equilibrium adsorption constant **K**, g/t	17,731
Product of kinetic and equilibrium adsorption constants **kK**, h^{-1}	260

For the Fekola HG domain composite sample, the adsorption kinetics (k = 0.015 h^{-1}) are relatively slow but the equilibrium loading (K = 17,731) is favourable for this sample. Overall, the product of these two constants is 260 h-1 and shows very good adsorption properties. Based on the absence of any preg robbing characteristics and very good adsorption properties, CIP has been selected for the Fekola processing plant. The SGS Lakefield modelling study indicated excellent performance of CIP for Fekola soluble gold recovery.

13.3.15 Slurry Rheology Testwork

Rheology testwork was conducted on three metallurgical domain composite leach thickener underflow samples and one cyanide destruction tailings sample. The results are summarised below in Table 13.3.16.

Table 13.3.16 Summary of Fekola Slurry Rheology

Sample Characterization								[5] Bingham Plastic parameters						Observations
								Unsheared Sample			Sheared Sample			
Test Code	K_{80} microns	[1] ASG g/cm³	[2] SG g/cm³	[3] α ASG/SG	pH ---	Solids % w/w	[4] Dens. g/L	Shear Rate ([6] γ) range, 1/s	Yield Stress (τ_{yB}) Pa	Plastic Viscosity (η_P) mPa.s	Shear Rate ([6] γ) range, 1/s	Yield Stress (τ_{yB}) Pa	Plastic Viscosity (η_P) mPa.s	
Sample: Hanging Wall Domain Comp underflow								[7]CSD = ~71% solids, corresponding to ~ 32 Pa unsheared and 28 Pa sheared yield stress.						
T2	74	2.73	2.81	0.97	10.4	75.0	1908	240 - 480	121	335	240 - 480	93	282	Thixotropic
T1		2.72		0.97		73.1	1860	240 - 480	63	142	240 - 480	51	128	Thixotropic
T3		2.73		0.97		70.1	1799	240 - 480	18	55	240 - 480	19	50	None
T4		2.73		0.97		66.0	1719	240 - 480	5.7	25	240 - 480	6.8	24	None
T5		2.73		0.97		61.0	1631	240 - 480	1.7	14	240 - 480	2.4	16	Minor settling. Dilatant >480 1/s
T6		2.69		0.96		45.4	1400	20 - 150	0.3	4	20 - 150	0.2	4	Minor settling during measurement.
Sample: Low Grade Domain Comp underflow								[7]CSD = ~71% solids, corresponding to ~ 36 Pa unsheared and 31 Pa sheared yield stress.						
T7	79	2.75	2.84	0.97	10.3	75.4	1923	240 - 480	157	460	240 - 480	133	322	Thixotropic
T8		2.75		0.97		73.8	1886	240 - 480	106	179	240 - 480	87	148	Thixotropic
T9		2.76		0.97		70.9	1824	240 - 480	36	61	240 - 480	33	56	Thixotropic
T10		2.76		0.97		66.6	1737	240 - 480	12	27	240 - 480	11	25	None
T11		2.73		0.96		60.6	1623	240 - 480	2.9	14	240 - 480	3.2	14	None
T12		2.71		0.95		45.3	1401	20 - 150	0.3	5	20 - 150	0.3	4	Minor settling during measurement.
Sample: High Grade Domain Comp underflow								[7]CSD = ~72% solids, corresponding to ~ 26 Pa unsheared and 21 Pa sheared yield stress.						
T13	81	2.77	2.83	0.98	10.7	75.6	1933	240 - 480	107	291	240 - 480	59	257	Thixotropic
T14		2.76		0.97		74.2	1898	240 - 480	64	150	240 - 480	41	133	Thixotropic
T15		2.76		0.97		72.0	1848	240 - 480	24	73	240 - 480	20	63	Thixotropic
T16		2.77		0.98		67.6	1762	240 - 480	6.1	29	240 - 480	5.9	27	None
T17		2.76		0.97		62.2	1657	240 - 480	1.5	14	240 - 480	1.6	17	None
T18		2.72		0.96		44.8	1395	20 - 150	0.1	4	20 - 150	0.1	4	Minor settling during measurement.
Sample: CND Tailing (HG Domain Comp) underflow								[7]CSD = ~73.5% solids, corresponding to ~ 36 Pa unsheared and 28 Pa sheared yield stress.						
T23	71	2.77	2.82	0.98	8.1	76.7	1962	300 - 500	114	413	300 - 500	81	319	Thixotropic
T19		2.79		0.99		74.8	1921	200 - 400	58	193	200 - 400	43	166	Thixotropic
T24		2.81		1.00		73.9	1904	200 - 400	42	124	200 - 400	32	112	Thixotropic
T20		2.81		1.00		72.7	1882	200 - 400	26	89	200 - 400	22	77	Thixotropic
T21		2.79		0.99		70.7	1830	200 - 400	13	51	200 - 400	12	45	None
T22		2.80		0.99		67.7	1770	200 - 400	5.5	27	200 - 400	5.3	25	None
T25		2.79		0.99		45.4	1411	20 - 150	0.1	3	20 - 150	0.02	3	Moderate settling during measurement.

All samples were tested at ambient temperature (~19-22°C).
All domain comp underflow samples contained BASF Magnafloc 1011 flocculant at a dosage of 25 g/t.
CND Tailing contained BASF Magnafloc 1011 flocculant at a dosage of 30 g/t.
The values are based on data produced by the unsheared and sheared slurry sample.
Variable shearing was produced in the 0 to 600 1/s range, increasing and decreasing (up and down curves).
Constant shearing was produced by subjecting the slurry sample to a constant rotation at 300 1/s for 180 seconds.
[1] ASG - Actual specific gravity of dry solids, calculated from the measured pulp density ([4] Dens.).
[2] SG - Dried solids specific gravity, measured with the pycnometer
[3] α - Correction Factor, i.e. solids density ratio.
[4] Pulp density in g/L.
[5] Bingham Plastic parameters: yield stress (τ_{yB}) and plastic viscosity (η_P) values, for the specified γ range.
[6] Shear rate range at which the rheological parameters were calculated.
[7] The Critical Solids Density (CSD) value is predictive of the maximum underflow solids density achievable in a commercial thickener and of the underflow solids density and pumpability ranges achievable in practice and with reasonable friction pressure losses for an economically feasible operation.

The results indicate the following:

- The samples exhibited low apparent viscosity at low pulp densities, indicating that 'sanding' of the leach and CIP tanks may pose a problem if the circuit is operated at low pulp densities. The inclusion of a leach thickener ahead of the leach and CIP circuits will allow the slurry tanks to be operated at densities of 45% w/w solids or up to 50% solids for good slurry mixing characteristics.

- Pumping of solids at densities up to 70% w/w solids using centrifugal pumps should not be problematical.

13.3.16 Thickening and Flocculation Testwork

Samples of the three metallurgical domain composites, HG, LG and HW, along with a sample of cyanide destruction tailings were tested by technical representatives from FLSmidth in the SGS Lakefield laboratory. Testwork comprised of flocculant screening, static and dynamic thickening tests and thickened mud rheology evaluation.

Table 13.3.17 Summary of Thickening Testwork Results

Test Conditions	LGC	HGC	HWC	CND Tails
Recommended Diluted Feed Solids Concentration (wt%)	8	12	10	14
3Recommended Total Floc Dose MF1011 (g/t)	24	32	28	30
pH	8.6	8.1	8.9	8.9
RHEOLOGICAL CHARACTERISTICS				
Est. Bed Solids at 0.5 hr Retention Time (wt%)/ Est. Yield Stress	63.9 (<20 Pa)	50.6 (<20 Pa)	46.0 (<20 Pa)	-
Est. Bed Solids at 1 hr Retention Time (wt%)/ Est. Yield Stress	67.4 (<20Pa)	60.4 (<20 Pa)	56.1 (<20 Pa)	65.9 (<20 Pa)
Est. Bed Solids at 2 hr Retention Time (wt%) / Est. Yield Stress	69.6 (<20 Pa)	64.8 (<20 Pa)	62.6 (<20 Pa)	71.9 (<20 Pa)
Est. Bed Solids at 3 hr Retention Time (wt%) / Est. Yield Stress	71.0 (<20 Pa)	69.5 (<20 Pa)	65.5 (<20 Pa)	-
Est. Bed Solids at 6 hr Retention Time (wt%) / Est. Yield Stress	-	-	-	76.0 (90 Pa)
LEACH THICKENER SIZING				
Design Underflow Density (wt% Solids)	45	45	45	60
Recommended Unit Area (m^2/tpd)	0.03	0.03	0.03	0.03
Design U/F Yield Stress (Pa)	<20	<20	<20	<20

Salient outcomes of testwork are:

- Based on flocculent screening, MF1011 provides good settling characteristic. MF1011 is an anionic polyacrylamide flocculant with a high molecular weight and medium charge density.

- The optimum thickener feed density ranges from 8% to 14% as listed in Table 13.3.17. The thickener feedwell includes auto-dilution ports to achieve the desired thickener feed density.

- The results from the testwork indicate a thickener unit area at 0.03 m^2/tpd is required. For the maximum throughput of 5 Mtpa, this equates to a 24 m diameter thickener without any design margin. Allowance for additional mud residence time and increased operational flexibility of the equipment results in a design margin of 10% being added to the thickener sizing. Therefore, a high rate thickener with a diameter of 25 m will be selected for each duty based on these testwork results.

- The testwork results show that thickener underflow densities of 45% solids and 60 to 65% solids (w/w) can be readily achieved for the leach and tailings thickener duties respectively.

13.3.17 Materials Handling Testwork

Materials handling testwork has been conducted by Jenike & Johanson Limited (JJL), who were commissioned to perform flow property tests on three comminution domain composite samples, Deep HG, Shallow HG and LG. Based on the results obtained, Jenike and Johanson provided functional design recommendations for the crushed ore stockpile and reclaim system.

The testwork programme included the following tests:

- Particle density: completed using a 100 mls graduated cylinder.

- Compressibility: determines the bulk density vs. consolidating pressure relationship.

- Loose and Compacted Bulk Density: done using a 20-litre pail.

- Flow Function: for calculating the critical outlet dimensions.

- Wall Friction: for calculating mass-flow hopper angles.

- Critical Chute Angle: for calculating critical chute angle.

- The flow property testwork results are expressed in terms of bin dimensions required to ensure dependable flow, maximum hopper angles for mass flow, minimum chute angles and, critical discharge rates through bin outlets. Based on the results of the Fekola ore and experience from the design and operation of the Otjikoto processing plant, which also had a similar testwork programme conducted by JJL, the Fekola crushed ore stockpile and reclaim system will be similar to the Otjikoto design with some enhancements to further improve the live capacity and ore flow consistency.

13.4 Comminution Circuit Selection

The following section presents details on:

- The comminution circuit design basis.
- The comminution circuit modelling by Orway Mineral Consultants.

13.4.1 Comminution Circuit Design Basis

The comminution characteristics of the deposit are summarised in Table 13.3.13. The ore from the Fekola deposit is classified as hard to very hard competency with above average grinding energy requirements and is moderate to highly abrasive.

The crushing and grinding circuit design is based on treating 4 Mtpa of ore expandable to a maximum throughput of 5 Mtpa without any further design margin. The opportunity to incorporate an unused SAG mill into the comminution circuit has also been assessed during the comminution circuit modelling.

13.4.2 Comminution Circuit Modelling

Orway Mineral Consultants (OMC) were commissioned to model the physical data for the Fekola deposit and recommend the most suitable comminution circuit configuration and SAG / Ball mill size combination to achieve a P_{80} grind size of 75 microns. Based on their experience with similar projects and cross-referencing of their extensive in-house database, OMC have recommended a single stage crushing circuit followed by a SAG mill and ball mill with pebble crushing (SABC) circuit. The key design criteria used in modelling of the circuit configurations is summarised in Section 13.5.2.

Grinding circuit modelling was also completed by SGS Lakefield, who use a similar but different modelling method compared to OMC. SGS Lakefield also recommended a similar SABC circuit with mills of the same diameter, but different motor sizing and effective grinding length. OMC use a power based modelling approach, coupled with data from operating projects with comparable hard ore, the OMC modelling and mill sizing selection has been adopted. SGS Lakefield use the JKSimMet modelling approach.

The grinding circuit has been modelled using the ore specific physical parameters summarised in Section 13.5.2.

13.4.3 Comminution Circuit Selection

A gyratory crusher followed by a SAG and ball milling circuit with a pebble crusher has been adopted for the OFS as this approach provides significant capital and operating cost savings over the other comminution circuit options considered in previous feasibility studies.

OMC have also predicted the grinding media consumption rate, mills and pebble crusher wear liner consumption rates based on the average abrasion index as well as the average power consumption based on the average ore properties.

After OMC completed the mill size selection, an unused SAG mill of a larger diameter was found to be available and reviewed by the OMC model. A detailed assessment has shown that this unused SAG mill will operate with a lower ball charge and result in a reduced grinding media consumption rate. This lower operating cost will offset the increased capital installation expenditure required for this mill. Further details of the recommended grinding circuit configuration are detailed in Section 17 and Section 24.

13.5 Metallurgical Testwork Summary

13.5.1 Selected Treatment Route

The metallurgical treatment route selected has been based on the results of the current testwork programme and can be summarised as follows:

- Single stage primary crushing.
- SAG milling, ball milling and pebble crushing (SABC).
- Leach thickening, which offers the following benefits:
  - the ability to operate at low cyclone overflow densities in the grinding circuit thereby improving classification efficiencies which potentially improves overall gold recoveries and ball mill grinding efficiency
  - consistent and sufficiently high leach and CIP feed density to ensure adequate solids and carbon suspension
  - to separate gold bearing solution that has been dissolved in the grinding circuit for carbon adsorption recovery in the Carbon-in-Columns (CIC) circuit.
- Leach and Carbon-in-Pulp (CIP).
- Pressure Zadra stripping circuit.
- Cyanide destruction using SO_2 / Air.
- Tailings thickening, which offers the benefit of being able to recover water before the tailings slurry is discharged to the TSF. This will halve the fresh water requirement for the processing plant.

13.5.2 Key Process Design Parameters

The key process design parameters derived from testwork are listed in Table 13.5.1.

Table 13.5.1 Summary of Grinding, Leach and CIP Circuit Design Parameters

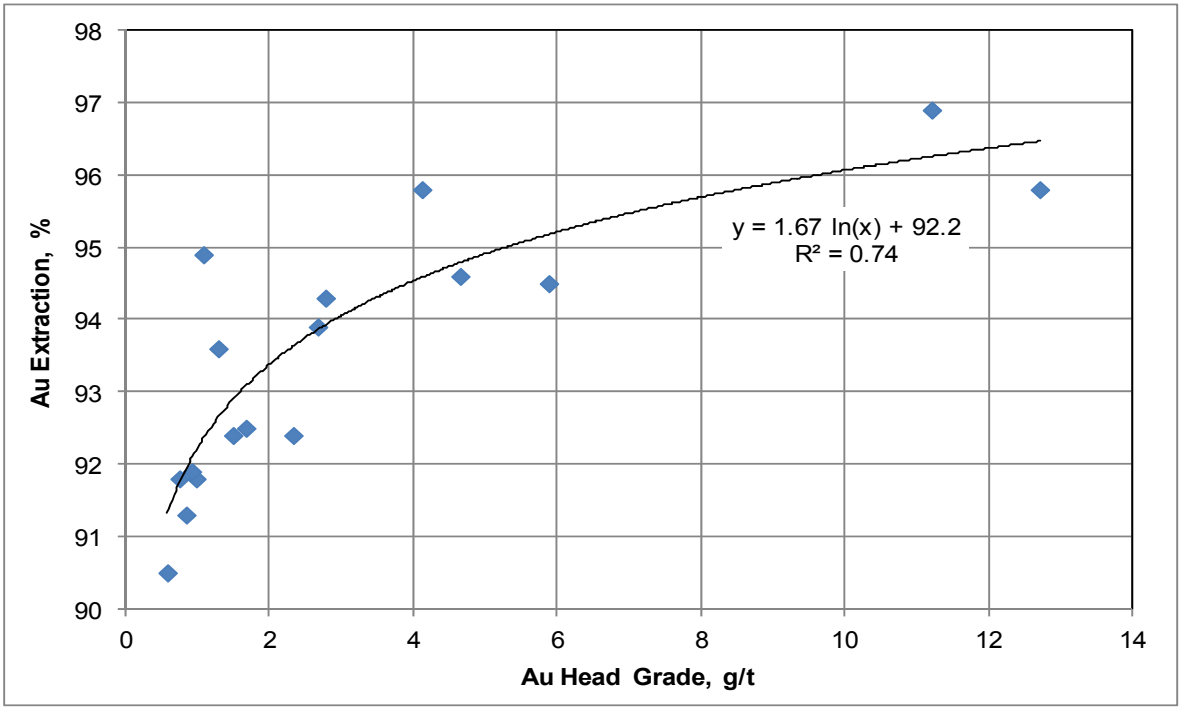
Parameter	Unit	Value
UCS	Mpa	106.9
Crushing Work Index	kWh/t	15.8
Bond Rod Mill Work Index	kWh/t	21.0
Bond Ball Mill Work Index	kWh/t	20.3
JK Drop Weight Index, A x b		27.3
SMC A x b		28.1
SMC ta		0.26
Crushing Circuit Product Size	mm, P_{80}	150
Milling Circuit Product Size	microns, P_{80}	75
Leach Thickener Specific Settling Rate	m^2/tpd	0.3
Pulp Density	% solids (w/w)	45
pH		10.5
Leach Residence Time	hours	24
CIP Residence Time	hours	6
Tailings Thickener Specific Settling	m^2/tpd	0.3

13.5.3 Metallurgical Recoveries

Gold Extraction Model

The metallurgical results of leach tests conducted on the master composite and variability samples at the optimum grind and cyanidation conditions are summarised in Table 13.3.13. Overall gold extractions range from 87 to 97%. The variability sample results show a logarithmic relationship with a strong correlation coefficient (r = 0.86) for the measured gold head grade and resulting gold extraction under optimised leach conditions at a grind size of 75 microns. The relationship is graphically illustrated in Figure 13.5.1. It should be noted that the test result from test CN-39 (FEK-MET-008, 87.3% gold extraction) has been omitted from the data set as it has been identified as an outlier. However, when this outlier result is included in the data set, the predicted gold extraction is reduced by 0.4 to 0.5%, which is not significant and within the plant recovery discount of 1.0%. It is recommended that further variability samples from future drilling are leached with optimised leaching conditions at a grind size of 75 microns to further confirm the gold extraction model.

Figure 13.5.1 Gold Extraction Model



The overall relationship between gold head grade and gold extraction is shown by the following relationship:

Gold Extraction (%) = 1.6705 ln (Gold Head Grade (g Au/t)) + 92.218 [A]

At a gold head grade of 2.50 g Au/t, the predicted gold extraction is 93.7%.

The metallurgical gold extraction is a statement of the anticipated metallurgical extraction for the Fekola deposit but is not a statement of the anticipated plant performance. Consideration must be given to soluble gold and fine carbon losses, potential for short circuiting in the leach circuit and other associated plant problems that may impact on the overall plant gold recovery.

Anticipated Plant Recovery

The gold extraction model represents laboratory conditions. As laboratory testwork is conducted under ideal conditions, the overall plant recovery prediction incorporates an allowance for the loss of recovery that will occur due to plant and operating inefficiencies and to account for soluble gold and fine carbon losses to tailings. The overall deficiency in the gold extraction model is estimated to be 1.0%.

The anticipated plant gold recovery is estimated as follows:

Plant Gold Recovery (%) = Gold Extraction (%) – 1.0% [B]

Where gold extraction is estimated by equation [A].

At a gold head grade of 2.50 g Au/t the estimated plant gold recovery is 92.7%.

Neither the gold extraction model nor the anticipated plant recovery model includes any benefit from the addition of a carbon column adsorption circuit that was included in the process flowsheet when the gravity circuit could no longer be justified based on the results of the gravity recoverable gold modelling. Since, the leach optimisation testwork shows pre-aeration of the leach feed slurry is not required, some sodium cyanide solution will be added to the SAG mill feed chute to start the leaching of gold in the grinding circuit. This means that after grinding, the leach thickener overflow solution will contain dissolved gold and the carbon column adsorption circuit will recover dissolved gold before entering the leach and CIP circuits. This means that gold can be recovered at the beginning of the overall flowsheet and facilitate early recovery of gold, particularly during high gold head grade periods.

13.5.4 Reagent Consumption

The metallurgical results of leach tests conducted on the master composite and variability samples at the optimum grind and cyanidation conditions are summarised in Table 13.3.13. The results indicate that the lime and sodium cyanide addition do not vary significantly.

The lime and sodium cyanide addition rates from the testwork have been proportioned according to the distribution of tonnes from each domain as per the mineral resource issued early 2015. The reagent addition values are presented in Table 13.5.2.

Table 13.5.2 Summary of Lime and Sodium Cyanide Addition

Reagent	Unit	Design Addition Rate
Sodium Cyanide	kg/t	0.78
CaO[1]	kg/t	0.67
Quicklime[2]	kg/t	0.74

1. Testwork conducted using reagent grade hydrated lime and addition reported as CaO.
2. Quicklime contains 90% CaO, based on supplier information.

13.5.5 Testwork Conclusions

The following conclusions can be drawn from the metallurgical and comminution testwork programmes:

- The Fekola deposit is classified as hard to very hard competency with above average grinding energy requirements and is moderate to highly abrasive. The ore is amenable to primary crushing followed by a SAG mill and ball mill with pebble crushing (SABC).
- Fekola ore is predominantly 'free-milling', not 'preg robbing' and is amenable to gold extraction by conventional cyanidation.

- A gravity separation circuit is not warranted for the Fekola deposit. Instead, a carbon column adsorption circuit will be included to recover dissolved gold leached in the grinding circuit to facilitate early recovery of gold, particularly during high gold head grade periods.

- The optimum leaching conditions identified are 24 hour cyanidation with 350 ppm NaCN, initial lead nitrate addition of 100 g/t, pH 10.3 to 10.5, dissolved oxygen levels of ~15 ppm and a pulp density of 45% solids (w/w). The addition of lead nitrate and dissolved oxygen levels of 15 ppm is found to be beneficial in leach kinetics and overall recovery. Anticipated lime and cyanide addition rates are moderate.

- The ore typically yields good recoveries (87 to 97%). Testwork results show a logarithmic relationship between the measured gold head grade and resulting gold extraction under optimised leach conditions at a grind size of 75 microns.

- Based on the absence of any preg robbing characteristics and very good adsorption properties, a CIP circuit has been selected for the Fekola process flowsheet.

- The cyanidation tailings responded well to cyanide destruction treatment using the SO_2 / Air process.

- The ore has a thickener specific settling rate of 0.3 m^2/tpd for both the leach and tailings thickener duties. The leach thickener requires 32 g/t of flocculant to produce an underflow with a density of 45% solids (w/w). The tailings thickener requires 30 g/t of flocculant to produce an underflow with a density of 60 to 65% solids (w/w).

Table of Contents

			Page
14.0	**MINERAL RESOURCE ESTIMATE**		**14.1**
	14.1	Summary	14.1
	14.2	Software Used	14.2
	14.3	Available Data	14.2
	14.4	Exploratory Data Analysis	14.3
	14.5	Geological Interpretation and Modelling	14.4
		14.5.1 Structural Model	14.5
		14.5.2 Lithological and Pyrite Models	14.5
		14.5.3 Weathering Domains	14.5
		14.5.4 Mineralization Domains	14.8
		14.5.5 Contact Analysis	14.15
		14.5.6 High Grade Shoot	14.16
	14.6	Evaluation of Outliers, Compositing and Variography	14.16
		14.6.1 Evaluation of Outliers	14.16
		14.6.2 Composites	14.17
		14.6.3 Variography	14.17
	14.7	Resource Block Model	14.21
		14.7.1 Block Model Dimensions	14.21
		14.7.2 Gold Grade Estimation	14.22
		14.7.3 Discussion of Re-blocking	14.22
		14.7.4 Bulk Density	14.23
	14.8	Model Validation	14.23
		14.8.1 Visual Comparison of Composites to Block Grade Estimates	14.23
		14.8.2 Comparison Statistics	14.26
		14.8.3 Swath Plots	14.26
		14.8.4 Comparison to Previous Resource Models	14.29
	14.9	Mineral Resource Classification	14.31
		14.9.1 Reasonable Prospects of Eventual Economic Extraction	14.31
	14.10	Mineral Resource Statement	14.31
		14.10.2 Sensitivity of Mineral Resource to Cut-off Grade	14.32
	14.11	Recommendations	14.33
	14.12	Comments on Section 14	14.33

TABLES

Table 14.1.1	Fekola Mineral Resources, as at 24 January 2015	14.2
Table 14.3.1	Drillhole Summary, Number and Meterage Used in Resource Model	14.3
Table 14.4.1	Statistics by Logged Percent Pyrite	14.3
Table 14.5.1	Weathering Domain Grouping	14.6
Table 14.5.2	Weathering Domain Codes	14.8
Table 14.5.3	Mineralization Domain Codes	14.8
Table 14.5.4	Gold Assay Statistics by Mineralization Domain Codes for Fresh Rock	14.13
Table 14.5.5	Gold Assay Statistics by Mineralization Domain Codes for Saprolite	14.14
Table 14.6.1	Mineralization Domain Codes	14.17
Table 14.6.2	Capped 2 m Gold Grade Composite Statistics by Mineralization Domain	14.17
Table 14.6.3	Variogram Models, All Mineralization Domains	14.18
Table 14.6.4	Variogram Models, Domain 101	14.18
Table 14.6.5	Variogram Models, Domain 105	14.19
Table 14.6.6	Variogram Models, Domain 110	14.19

Table of Contents (Continued)

		Page
Table 14.6.7	Variogram Models, Domain 125	14.20
Table 14.7.1	Block Model Specifications	14.21
Table 14.7.2	Block Model Fields	14.21
Table 14.7.3	Gold Grade Estimation Plan	14.22
Table 14.7.4	Densities used in Tonnage Calculations	14.23
Table 14.8.1	Comparison of Estimation Means at 0 g/t Cut-off	14.26
Table 14.8.2	Comparison of Modelling Methodologies, MPR August 2013 and B2Gold's February 2015 Model	14.30
Table 14.8.3	Comparison of B2Gold March 2015 Model to MPR August 2013 Model	14.30
Table 14.9.1	Parameters used for Resource Pit Optimization Runs	14.31
Table 14.10.1	Fekola Mineral Resources, as at 24 January 2015 [1,2,3,4,5,6]	14.32
Table 14.10.2	Sensitivity Analysis to Mineral Resources by Gold Cut-off Grades, as at 24 January 2015	14.32

FIGURES

Figure 14.4.1	Gold Grade Distributions by Logged Lithology	14.4
Figure 14.5.1	Cross Section Looking North, Mineralization Domains	14.9
Figure 14.5.2	Longitudinal Section Looking West, Mineralization Domains	14.9
Figure 14.5.3	Level Plan, Mineralization Domains	14.10
Figure 14.5.4	Statistics of Logged Percent Pyrite by Mineralization Domain	14.11
Figure 14.5.5	Statistics of Gold Grade by Mineralization Domain	14.12
Figure 14.5.6	Contact Plot between Domain 101 and 105	14.15
Figure 14.5.7	Contact Plot between Domain 105 and 110	14.15
Figure 14.5.8	Contact Plot between Domain 110 and 125	14.16
Figure 14.8.1	Cross Section Looking North, Block Model and Composite Gold Grades	14.24
Figure 14.8.2	Plan View, Elevation 0 m, Block Model Gold Grades	14.25
Figure 14.8.3	Comparison of Grade Estimates by Northing	14.27
Figure 14.8.4	Comparison of Grade Estimates by Elevation	14.28
Figure 14.8.5	Comparison of Grade Estimates by Easting	14.29

14.0 MINERAL RESOURCE ESTIMATE

14.1 Summary

Mineral resources for the Fekola deposit are reported from B2Gold's mineral resource model which has an effective date of 24 January 2015. This updated mineral resource estimate was completed in February 2015 and replaces the previous two mineral resource estimates completed by MPR Geological Consultants Pty. Ltd. in January 2013 (MPR, January 2013) and August 2013 (MPR, August 2013). These previous estimates were reported by B2Gold in "Fekola Project, Mali, NI43-101 Technical Report on Preliminary Economic Assessment" dated August 2014 (B2Gold, 2014).

The B2Gold February 2015 model update incorporates drilling results available at 24 January 2015, revised geological logging and a new geological understanding on mineralization controls which is reflected in the interpretations and wireframes serving as the basis for grade estimation. The estimation methodology and drill spacing requirements for resource categories were also revised.

Mineralization domains at nominal grade thresholds of 0.1 g/t (domain 101), 0.5 g/t (105), 1.0 g/t (110), and 2.5 g/t (125) were implicitly modelled as 3D solids using Leapfrog software. The overall interpretation and dimensions of the mineralization domains were controlled by zones of high pyrite which were superimposed on the lithology model which considered regional folding and faulting. Assays were capped by mineralization domain and by fresh or saprolite prior to compositing to 2 m downhole intervals. Capping levels for fresh rock are 1.5, 2.8, 9.0, and 30 g/t for domain 101, 105, 110, and 125, respectively. In saprolite, the capping levels are 1.0, 2.1, 5.0 and 10 g/t for the same domains. The lower capping levels in saprolite are a function of the higher grade variability observed in saprolite assays.

Mineralization domain wireframes were coded to the blocks using sub-cells (minimum sub-cell 2.5 x 5 x 2.5 m) and the coded blocks were used as grade estimation domains. Gold grades were estimated into parent blocks (5 x 20 x 10 m) using Ordinary Kriging (OK) from 2 m capped composites for each domain. The sub-cell model was re-blocked to 5 x 10 x 5 m whole blocks to account for the gradational change in gold grades seen at the domain contacts. Variography confirmed observations seen on plots and in 3D that the main direction of grade continuity for domains 105 and 110 is azimuth 347° to 351°, with a shallow dip ranging from -10° to -14°. Block grade estimates were validated by visual comparison to composite grades, comparison of global block statistics to the nearest neighbour (NN) model, swath plots to check for local bias, and comparison to the previously reported mineral resource.

Mineral Resource classification was assigned to parent blocks. Indicated resources required an approximate drill spacing of 40 x 40 m and Inferred resources are drilled at 80 x 80 m. Average density by mineralization domain, which ranges from 2.76 to 2.83 gm/cm^3, was used for tonnage calculations. Saprolite and overburden densities were set at 1.8 gm/cm^3.

Reasonable prospects of economic extraction have been applied by reporting Mineral Resources from the re-blocked model above a gold cut-off grade of 0.6 g/t, within a conceptual pit shell run at a gold price of US$ 1,500/oz and preliminary operating costs as of March 2015.

This mineral resource block model is the basis for the mining studies (including pit optimizations and mining schedules) completed as part of the OFS.

Mineral Resources for Fekola, as at 24 January 2015, are reported from B2Gold's February 2015 block model and are shown on Table 14.1.1.

Table 14.1.1 Fekola Mineral Resources, as at 24 January 2015 [1,2,3,4,5,6]

Au Cutoff Grade (g/t)	Indicated			Inferred		
	Tonnes x 1000	Au g/t	Ounces x 1000	Tonnes x 1000	Au g/t	Ounces x 1000
0.6	61,574	2.16	4,281	9,055	1.68	490

(1) **Mineral Resources are estimated using best practices as defined by the CIM and reporting of Mineral Resources is compliant and in accordance with the disclosure requirements of NI 43-101.**
(2) **The Qualified Person for the Mineral Resources is Tom Garagan, P. Geo.**
(3) **Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.**
(4) **Mineral Resources are inclusive of Mineral Reserves.**
(5) **Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an indicated or measured Mineral Resource as a result of continued exploration.**
(6) **Mineral Resource numbers have been rounded to reflect the accuracy of the estimate and numbers may not add due to rounding.**
(7) **Mineral Resources are reported above a 0.6 g/t gold grade cut-off and within a conceptual pit shell using a gold price of US$ 1,500/oz and preliminary operating costs as of March 2015.**
(8) **Mineral Resources are reported on a 100% ownership basis.**

Recommendations with respect to the Mineral Resource estimate include additional drilling to confirm mineralization controls and continuity of high grade in the hanging wall zone, drilling at grade control spacing to assess the short range variability anticipated during mining and using other non-linear estimation techniques, such as conditional simulation to confirm the grade tonnage curve.

14.2 Software Used

The project drillhole database is compiled in MS-Access and imported to mining software packages via CSV (comma delimited ASCII) files. Wireframe models (solids models and surfaces) for mineralized domains, topography, overburden and saprolite were built using Leapfrog V2.1.2, Gems V6.6 and Surpac V6.6.2 software. Snowden Supervisor V8.3.1.2 was used for most statistical analyses and variography. B2Gold proprietary software, GSLIB and MS-Excel were also used for statistical analyses. Block model creation, grade estimation, block regularization and aspects of tabulations were done in Datamine V3.24.25.0. Pit optimization was run using Whittle V4.5.5 software.

14.3 Available Data

The database used as the basis for the mineral resource block model was compiled by B2Gold's Exploration staff at the project site from the previous owner's database (see Section 12, *Data Verification*), with review by B2Gold's Vancouver-based technical staff. The assay and geological database cutoff date was January 24, 2015 (filename "B2PapExport Med DH only Jan 24 2015 MJG"). All drillhole collar surveys are in UTM coordinates. All RC, DDH and RC / DDH holes within the block model limits were used in the estimate. Table 14.3.1(A) shows the number and

meterage of holes used to define the mineralization domains and Table 14.3.1 (B) shows the number of holes and meterage used directly in block grade estimation.

Table 14.3.1 Drillhole Summary, Number and Meterage Used in Resource Model

(A) Drillholes used in Interpretations of Mineralization Domains (Resource TAGS 100-125)

Drillhole Type	Number of Drillholes	Total Meters
DDH	172	51,267
RC	283	37,457
RC-DDH	39	13,490
Grand Total	494	102,214

(B) Drillholes used in Grade Estimation (Resource TAGS 101-125)

Drillhole Type	Total Drillholes	Total Meters
DDH	172	51,267
RC	267	36,104
RC-DDH	39	13,490
Grand Total	478	100,861

14.4 Exploratory Data Analysis

Exploratory data analysis (EDA) was done on the complete assay and logged database to better understand which logged geological characteristics have the strongest control on gold mineralization and to provide input to the re-logging process.

The logged percent pyrite data was binned and statistics (Table 14.4.1) were run on gold grades by the binned groups. The table shows with an increase in logged pyrite there is an increase in gold grade.

There is also a slight positive relationship between gold grade and increased silica and a similar but lesser correlation between gold grade and sericite alteration.

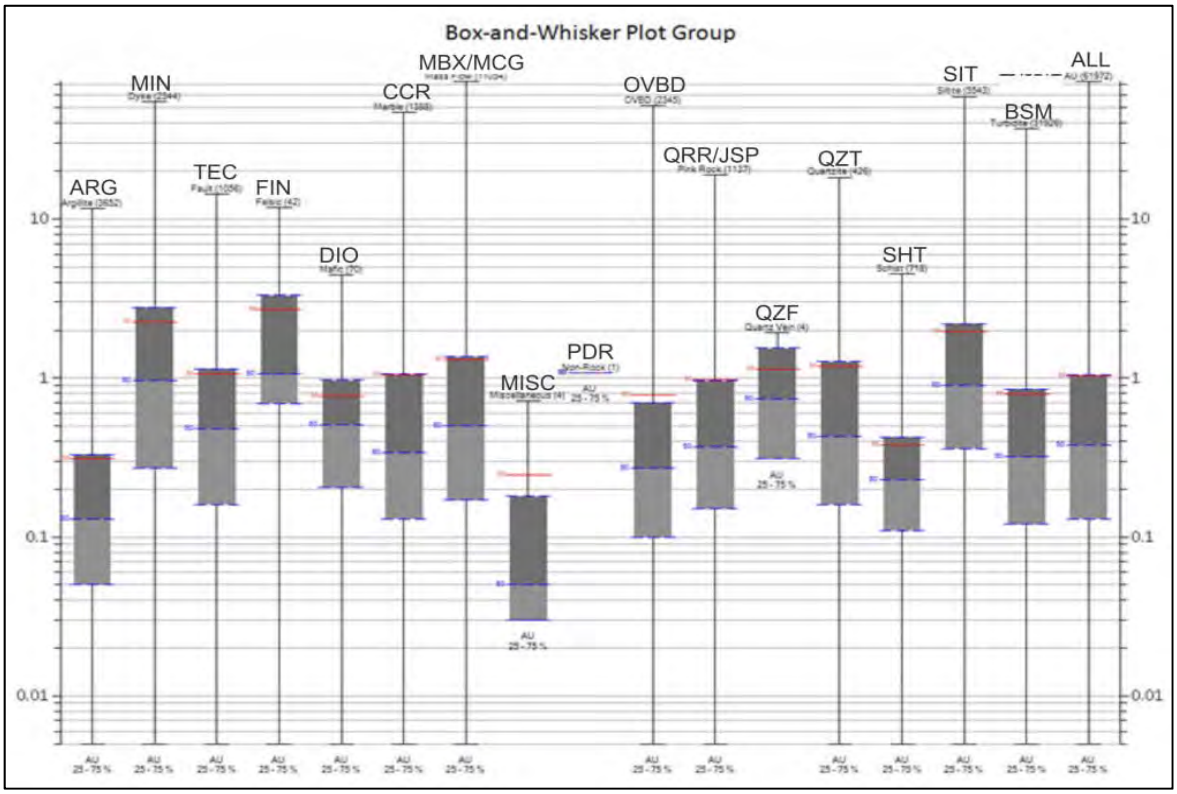
Table 14.4.1 Statistics by Logged Percent Pyrite

%Pyrite Bins	Bin	Number	Mean Au (g/t)	CV[(1)]	Min. Au (g/t)	Lower Quartile (g/t)	Upper Quartile (g/t)	Max [(2)] Au (g/t)
0	0	2,604	0.28	3.3	0.01	0.01	0.16	19.0
<1%	1	1,251	0.14	2.3	0.01	0.01	0.11	4.69
1 to <2%	2	15,750	0.29	2.7	0.01	0.01	0.25	19.0
2 to <5%	3	25,906	1.13	1.9	0.01	0.09	1.18	19.0
>=5%	4	1,269	2.95	1.3	0.01	0.47	3.92	19.0
-1 (not logged)	-1	16,593	--	--	--	--		--

Notes: [(1)] CV=coefficient of variation=standard deviation/mean, [(2)] Au capped at 19.0g/t for this exercise

Other logged parameters including lithology, vein types, structure, alteration, texture, grain size and sulphides do not show a strong relationship to gold mineralization. Gold grade statistics by logged lithology are shown on the boxplot on Figure 14.4.1. There isn't a strong control on gold grade by lithology although five lithologies (BSM, SIT, MBX, MCG and MIN) contain about 90% of the gold mineralization. Argillite (ARG) is commonly found in the project area; it is typically low grade and occurs at the periphery of higher grade mineralization.

Figure 14.4.1 Gold Grade Distributions by Logged Lithology



Box-and-whisker plot is a graphical display of the gold grade distribution by groupings (e.g., lithology). The red line is the mean, blue line is median, lower (25%) and upper (75%) quartiles shown as a rectangle, and the maximum and minimum noted with vertical lines.

Lithology abbreviations: ARG – Argillite; BSM – Siltstone/Mudstone; CCR- Dolomitic Mylonite; DIO- Mafic Igneous; FIN – Felsic Intrusive; JSP - Intense Alb-qtz-carb-hem alteration; MBX - Mass Flow Deposit; MCG - Meta-conglomerate; MIN – Mafic Intrusive; OVBD – Overburden; PDR- Non-Rock; QRR – Intense alb-qtz-carb-hem alteration; QZF – Quartz vein; QZT – Quartzite; SHT- Schist; SIT – Intense dol-ser-py alteration; TEC-Fault Gouge

14.5 Geological Interpretation and Modelling

Structural, lithological and pyrite interpretations / solid models were built iteratively. A preliminary structural model was built followed by a lithological model and pyrite model. Certain marker lithologies and pyrite supported modifying the structural model. This method allowed each model to inform the other models. The mineralization domain model was informed by the structural, lithological and pyrite models which ensured their basis was supported by observed geological features.

14.5.1 Structural Model

The main features of the structural model are the shallowly north-northwest plunging (13 to 354°) axis of the major F2 folding event and the late-stage, brittle Fekola Fault zone which cross cuts earlier generations of structures and mineralization. The Fekola Fault exhibits steep (70 to 80°) westerly dips in the north end of the deposit, with more moderate dips (45 to 50°) along strike to the south. The Fekola fault is a late structural feature with no significant influence on the geometry of mineralization. (Refer to Figure 7.3.5 in this report for an example cross section showing structural and lithology interpretations).

Folding and faults, as modelled, are based on foliation measurements in the geotechnical structural log and “FTZ” codes from the structure field in the DHgeology table.

14.5.2 Lithological and Pyrite Models

Based on field observations, supported by statistical and spatial reviews, logged lithologies were grouped for modelling as follows:

- Argillite.
- Turbidite (includes logged quartzite, quartz vein and schist).
- Mass flow breccia (which includes MBX, MGC, SIT, pink rock, mafic, felsic, dyke).
- Overburden and saprolite were also modelled.

At the deposit scale, grouped lithological units exhibit asymmetric, east-verging folds with local structural thickening of individual units at fold hinges and corresponding attenuation along fold limbs. The three lithological groups were modelled in 3D using Leapfrog software. The fold and fault orientations from the structural model were used to inform the lithology model.

Logged percent pyrite (py_pct) (where logging was available) was the basis for the pyrite interpretations and solids model. The structural and lithology models were iteratively used to inform the construction of the pyrite model.

14.5.3 Weathering Domains

Overburden, saprolite and fresh rock surfaces were modelled (“other” was not modelled) using logged lithology groups as shown on Table 14.5.1.

Table 14.5.1 Weathering Domain Grouping

LITH CODE	Description-from Original Logging	Group	Model Group
ARG	Mudstone	Argillite	Fresh
ARK	Arkose -Mafic dyke	Dyke	Fresh
MIN	Mafic intrusive	Dyke	Fresh
TEC	Fault gouge	Fault	Fresh
BRC	Fault Breccia	Fault	Fresh
CTC	Cataclasite-Fault Breccia	Fault	Fresh
PGR	Porphyritic granite	Felsic	Fresh
GRD	Granodiorite	Felsic	Fresh
GRA	Granodiorite	Felsic	Fresh
FIN	Felsic intrusion	Felsic	Fresh
RHY	Rhyolite	Felsic	Fresh
DOL	Dolerite	Mafic	Fresh
DIO	Diorite	Mafic	Fresh
MCD	Microdiorite	Mafic	Fresh
BAS	Mafic Volcanic	Mafic	Fresh
CCR	Marble -Dolomitic mylonite	Marble	Fresh
MFD	Mass Flow Deposit- complete cycle	Mass Flow	Fresh
MCG	Mass Flow Deposit-conglomerate	Mass Flow	Fresh
CNG	Conglomerate	Mass Flow	Fresh
GRW	Greywacke	Mass Flow	Fresh
MBX	Mass Flow Deposit - Breccia	Mass Flow	Fresh
MGW	Greywacke	Mass Flow	Fresh
SST	Sandstone	Mass Flow	Fresh
PEL	Pelite	Miscellaneous	Fresh
MYL	Mylonite	Miscellaneous	Fresh
TRM	Tourmalinite	Miscellaneous	Fresh
JSP	“Jasperoid”-Intense alb-qtz-carb-hem alteration	Pink Rock	Fresh
QRR	“QR Rock”Intense alb-qtz-carb-hem alteration	Pink Rock	Fresh
QZF	Quartz vein	Quartz Vein	Fresh
QZT	Quartzite	Quartzite	Fresh
QTT	Quartzite with tourmaline	Quartzite	Fresh
SHB	Biotite schist	Schist	Fresh
SHT	Schist	Schist	Fresh
SIT	“Siltite” -Intense dol-ser-py alteration	Siltite	Fresh
BSM	Banded Siltstone/Mudstone	Turbidite	Fresh
INC	Unknown rock	Non-Rock	Other
PDR	No Recovery -Non-Rock	Non-Rock	Other
GAL	Alluvial Gravel (Paleo-gravel)	ALV	OVB
ALV	Alluvium	ALV	OVB
LAT	Transported laterite	LAT	OVB
LTP	Laterite insitu	LAT	OVB
GLA	Lateritic Gravel	LAT	OVB
COV	Colluvium	OVBD	OVB
OVB	Overburden	OVBD	OVB

LITH CODE	Description-from Original Logging	Group	Model Group
SOL	Soil	OVBD	OVB
SLT	Silt	OVBD	OVB
CLY	Clay	OVBD	OVB
SAR	Saprock	SAP	SAP
MCZ	Mottled clay	SAP	SAP
RES	Residual Soil	SAP	SAP
GOS	Gossan	SAP	SAP
PCZ	Plasmic Clay	SAP	SAP
SAP	Saprolite	SAP	SAP

"Quotations" indicated visually identifiable lithologies with regional mine nomenclature.

Material Types

The overburden model represents various transported materials that overlie in-situ weathered rock and fresh rock. In some cases this material is locally derived and may contain anomalous Au values, although typically it does not. At this time, different overburden units (gravels, laterite, alluvium, etc.) have not been modelled separately. The average thickness of overburden is 10 m.

The saprolite model represents in-situ weathered rock. Saprolite proximal to mineralized fresh rock may be mineralized. Saprolite includes residual soils, gossans, clays and less intensely weathered rock termed "saprock". The average thickness of saprolite is 5 m.

The fresh rock model includes all unweathered rock, both mineralized and un-mineralized. Most of the mineralization of interest is in fresh rock.

Procedure for Building Weathering Domain Models

For the purposes of the weathering models only, drillhole collars were pressed to the LiDAR topographic surface to alleviate localized conflicts between surveyed collars and surveyed topography. The overburden / saprolite and saprolite / fresh surfaces were built using logged geology. In areas beyond drilling, the surfaces were projected using the trend from drillhole data.

Occasionally, if very close spaced drilling had conflicting logging, one of the conflicting drillholes was not used in surface creation in order to make a valid / useable surface. This process has no material effect on the final model.

The weathering domain model covers an area well beyond the limits of the resource block model to assist with other project planning requirements. Local variability is not represented in the model beyond drilling; this is not considered an issue as there is limited variability in the area supported by drilling.

The weathering surfaces were used to build 3D solid models which were used for coding the block model as shown on Table 14.5.2.

Table 14.5.2 Weathering Domain Codes

Weathering Domain	Domain Code
Overburden	5000
SAP	4000
Fresh	3000

14.5.4 Mineralization Domains

Review of statistics by logged geological parameters identified the need for additional domaining for grade estimation. Mineralization domains at nominal grade thresholds of 0.1 g/t, 0.5 g/t, 1.0 g/t and 2.5 g/t were implicitly modelled as 3D solids using Leapfrog software. The overall interpretation and dimensions of the mineralization domains were controlled by zones of high pyrite which were superimposed on the lithology model which considered regional folding and faulting. An example mineralization domain cross section is shown on Figure 14.5.1. Figure 14.5.2 shows a long section and Figure 14.5.3 shows a level. The 3D solids were coded to the block model as listed in Table 14.5.3. The minimum sub-cell size used for coding blocks was 2.5 x 5 x 2.5 m.

Table 14.5.3 Mineralization Domain Codes

Mineralization Domain	Domain Code
<0.1 g/t	100
0.1 g/t to 0.5 g/t	101
0.5 g/t to 1.0 g/t	105
1.0 g/t to 2.5 g/t	110
> 2.5 g/t	125

Figure 14.5.1 Cross Section Looking North, Mineralization Domains

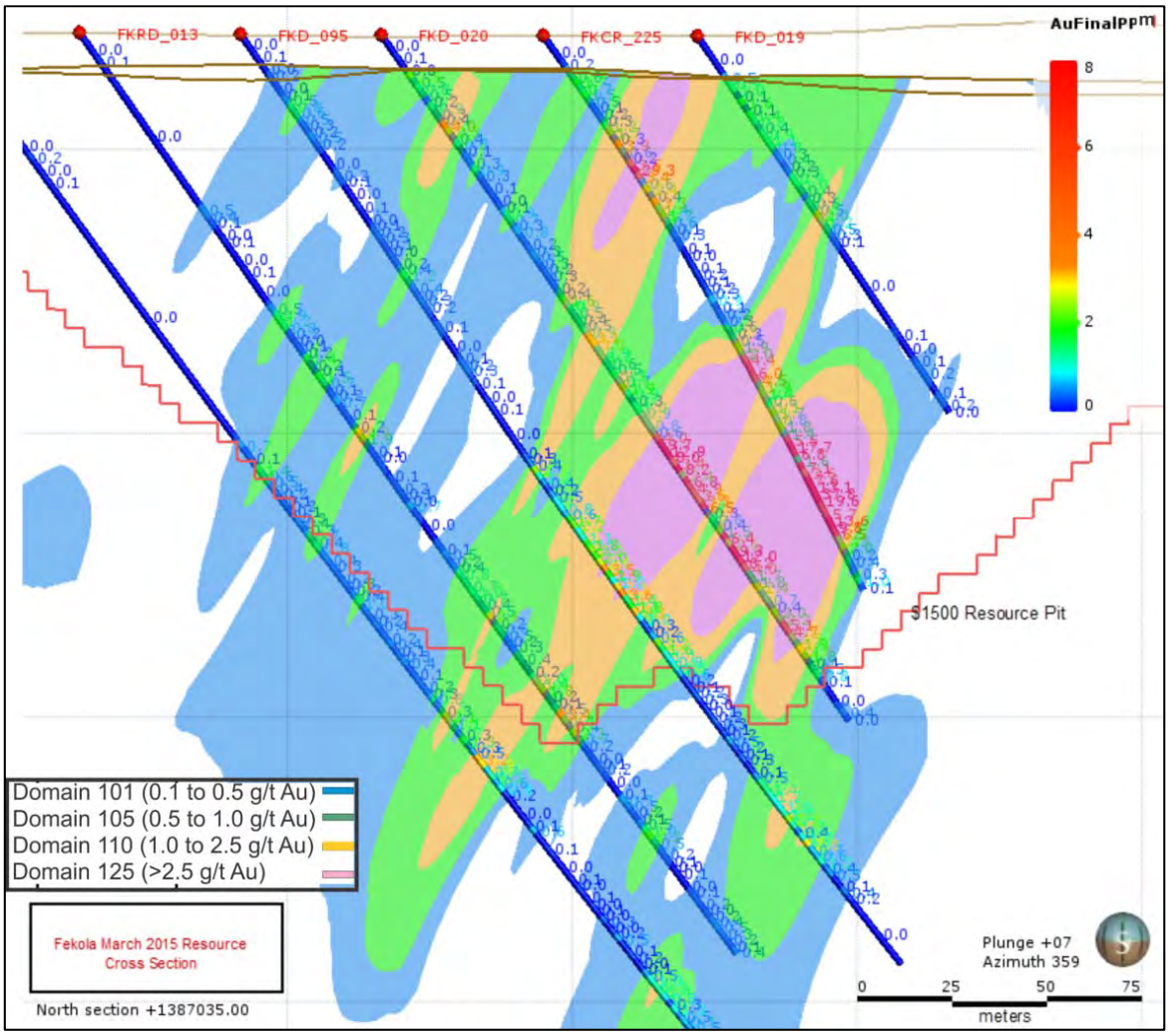


Figure 14.5.2 Longitudinal Section Looking West, Mineralization Domains

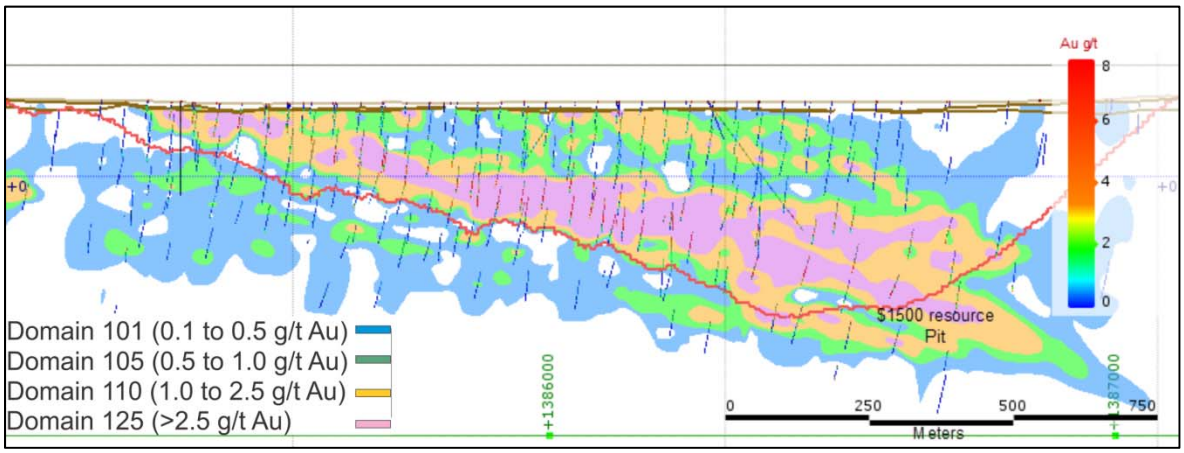
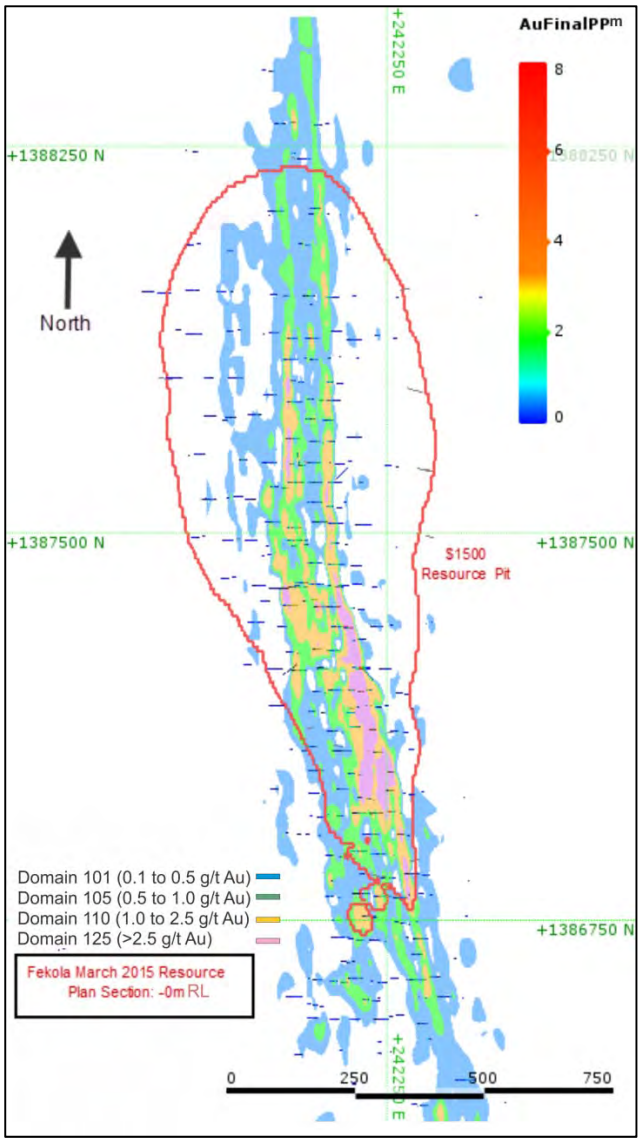


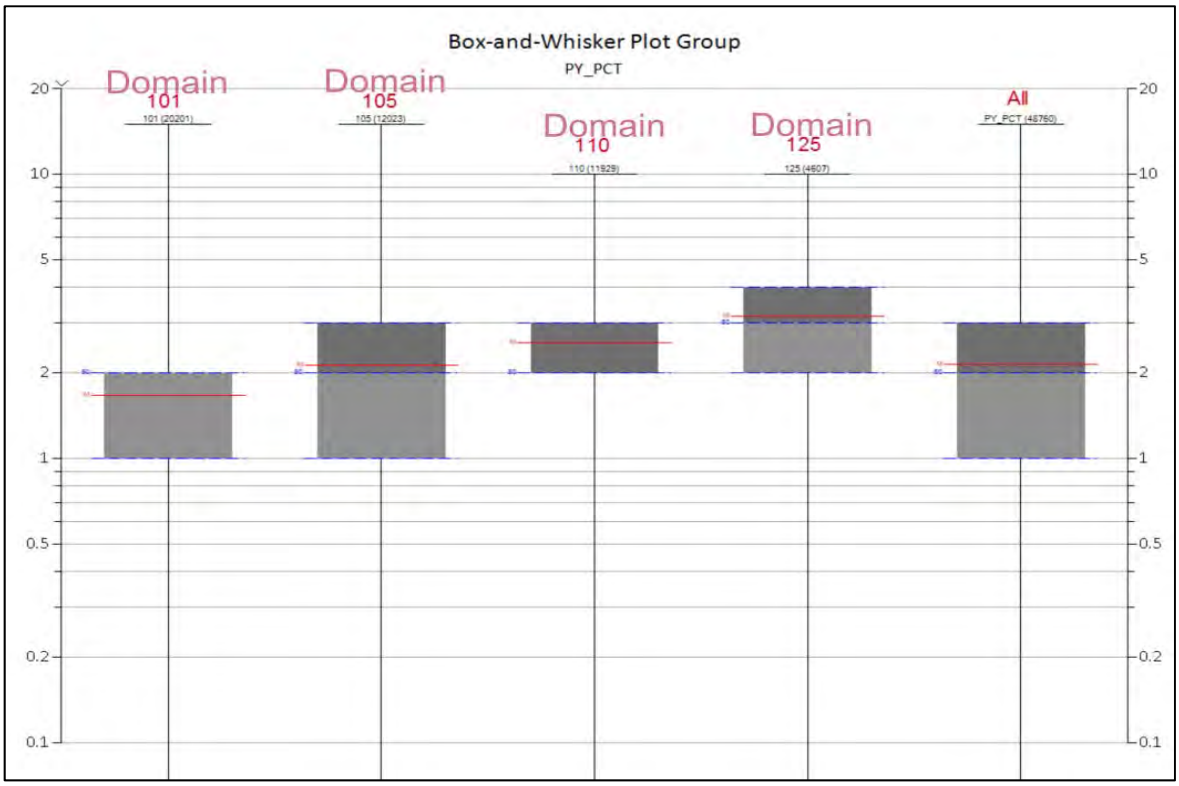
Figure 14.5.3 Level Plan, Mineralization Domains



Statistics by Modelled Mineralization Domains

EDA (described above) completed on gold assays by logged geological parameters showed increased gold grades with increased percent pyrite. By association the higher grade mineralization domains as modelled should have higher percent pyrite which is shown on Figure 14.5.4. The statistics for logged percent pyrite by modelled mineralization domains show higher averages and overall distributions of (logged) percent pyrite with the higher grade mineralization domains as modelled.

Figure 14.5.4 Statistics of Logged Percent Pyrite by Mineralization Domain



Gold grade statistics by mineralization domain are shown on Figure 14.5.5, Table 14.5.4 (fresh rock), and Table 14.5.5 (saprolite). Mineralization domains reasonably partition the grade distribution such that average gold grades are higher for the higher grade domains and the variability for each domain is relatively low.

Figure 14.5.5 Statistics of Gold Grade by Mineralization Domain

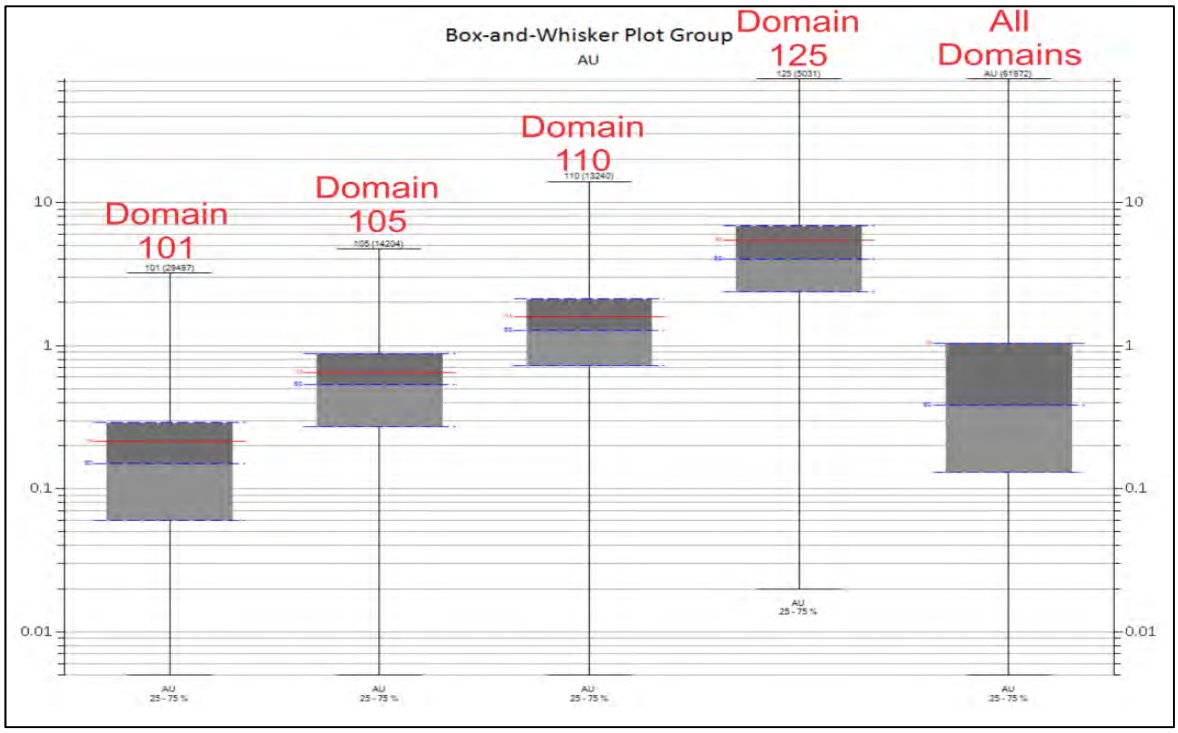


Table 14.5.4 Gold Assay Statistics by Mineralization Domain Codes for Fresh Rock

Gold Assays- Fresh	Uncapped	Uncapped	Uncapped	Uncapped	Capped	Capped	Capped	Capped
Cap Value					**1.5**	**2.8**	**9**	**30**
Domain	**101**	**105**	**110**	**125**	**101**	**105**	**110**	**125**
TYPE	**Fresh**	**Fresh**	**Fresh**	**Fresh**	**Fresh**	**Fresh**	**Fresh**	**Fresh**
Samples	26967	12932	12148	4483	26967	12932	12148	4483
Minimum	0.005	0.005	0.005	0.02	0.005	0.005	0.005	0.02
Maximum	3.22	4.71	14	73.2	1.5	2.8	9	30
Mean	0.214	0.64	1.573	5.409	0.214	0.638	1.571	5.337
Standard deviation	0.223	0.51	1.242	5.223	0.219	0.502	1.228	4.708
CV	1.042	0.797	0.789	0.966	1.027	0.787	0.781	0.882
Variance	0.05	0.26	1.542	27.275	0.048	0.252	1.507	22.165
10%	0.02	0.12	0.36	1.313	0.02	0.12	0.36	1.313
20%	0.05	0.22	0.6	2.05	0.05	0.22	0.6	2.05
30%	0.08	0.32	0.83	2.66	0.08	0.32	0.83	2.66
40%	0.11	0.42	1.06	3.3	0.11	0.42	1.06	3.3
50%	0.15	0.52	1.28	3.965	0.15	0.52	1.28	3.965
60%	0.19	0.64	1.55	4.81	0.19	0.64	1.55	4.81
70%	0.25	0.78	1.89	5.881	0.25	0.78	1.89	5.881
80%	0.34	0.98	2.34	7.75	0.34	0.98	2.34	7.75
90%	0.49	1.31	3.09	10.87	0.49	1.31	3.09	10.87
95%	0.65	1.63	3.86	14.4	0.65	1.63	3.86	14.4
97.50%	0.82	1.97	4.683	18.6	0.82	1.97	4.683	18.6
99%	1.04	2.4	6	25.5	1.04	2.4	6	25.5

Table 14.5.5 Gold Assay Statistics by Mineralization Domain Codes for Saprolite

Gold Assays-SAP	Uncapped	Uncapped	Uncapped	Uncapped	Capped	Capped	Capped	Capped
Cap Value					1	2.1	5	10
Domain	**101**	**105**	**110**	**125**	**101**	**105**	**110**	**125**
TYPE	**SAP**	**SAP**	**SAP**	**SAP**	**Sap**	**Sap**	**Sap**	**Sap**
Samples	1314	580	336	120	1314	580	336	120
Minimum	0.005	0.005	0.005	0.09	0.005	0.005	0.005	0.09
Maximum	1.48	3.52	6.99	51.7	1	2.1	5	10
Mean	0.207	0.627	1.448	5.8	0.205	0.622	1.424	4.204
Standard deviation	0.212	0.473	1.246	8.172	0.203	0.451	1.159	3.188
CV	1.025	0.754	0.86	1.409	0.992	0.725	0.814	0.758
Variance	0.045	0.224	1.552	66.784	0.041	0.203	1.344	10.163
10%	0.02	0.11	0.26	0.4	0.02	0.11	0.26	0.4
20%	0.05	0.23	0.512	1.13	0.05	0.23	0.512	1.13
30%	0.08	0.33	0.668	1.85	0.08	0.33	0.668	1.85
40%	0.11	0.44	0.85	2.72	0.11	0.44	0.85	2.72
50%	0.14	0.54	1.12	3.32	0.14	0.54	1.12	3.32
60%	0.18	0.66	1.45	4.32	0.18	0.66	1.45	4.32
70%	0.24	0.78	1.71	5.6	0.24	0.78	1.71	5.6
80%	0.33	0.97	2.166	7.22	0.33	0.97	2.166	7.22
90%	0.466	1.21	3.004	11.3	0.466	1.21	3.004	10
95%	0.65	1.52	3.878	20.6	0.65	1.52	3.878	10
97.50%	0.78	1.77	4.902	33.3	0.78	1.77	4.902	10
99%	1.017	2.138	6.066	39.54	1	2.1	5	10

14.5.5 Contact Analysis

Average gold grades by distance from the mineralization domain contact were plotted on 'contact plots' for the contact between domain 101 and 105 (Figure 14.5.6), domain 105 and 110 (Figure 14.5.7) and domain 110 and 125 (Figure 14.5.8). All plots show a 2 to 3 m transitional increase (or decrease, depending on direction) in grades when moving away from the contact, and not a sharp, distinct change in grade.

Figure 14.5.6 Contact Plot between Domain 101 and 105

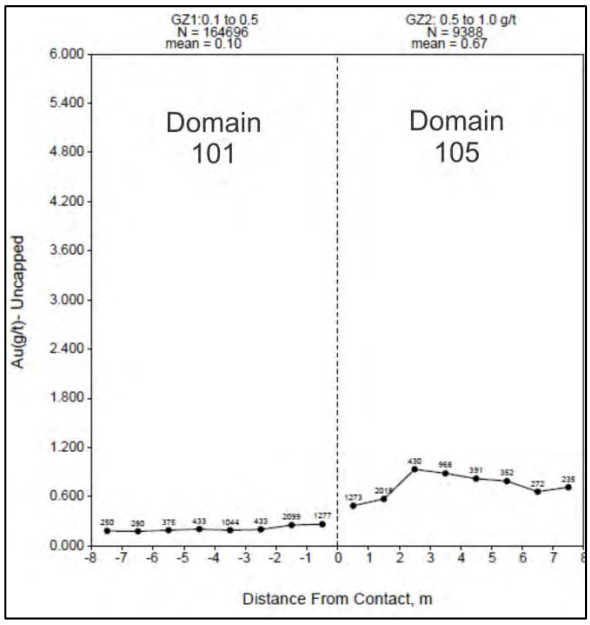


Figure 14.5.7 Contact Plot between Domain 105 and 110

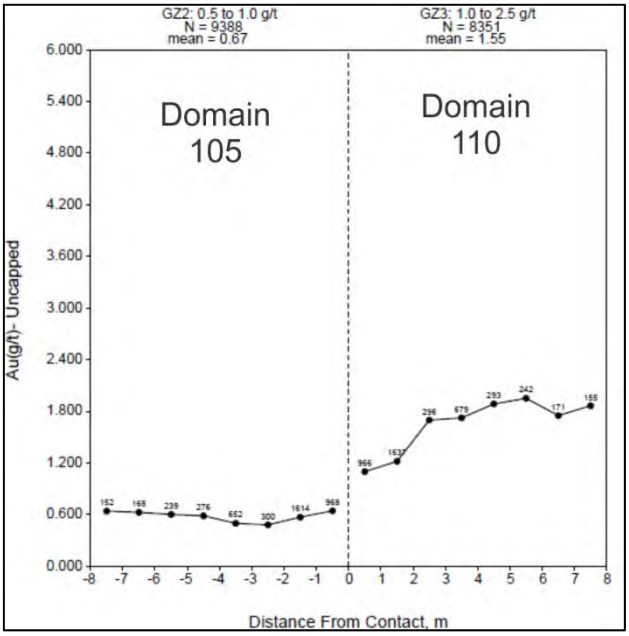
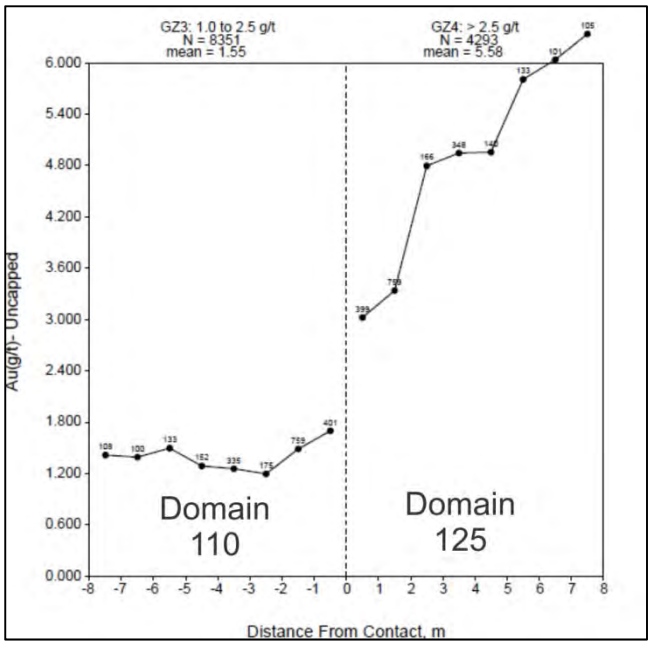


Figure 14.5.8 Contact Plot between Domain 110 and 125



14.5.6 High Grade Shoot

A solid model was built of the 'high grade shoot' which generally outlines the continuously mineralized area (Au >1.5 g/t) that strikes approximately N-S with a shallow plunge (-10° to -15°). The smaller high grade areas in the hanging wall are outside the high grade shoot. This solid was built to potentially allow application of different dilution scenarios, not for grade estimation. For this phase of work, this solid was not used.

14.6 Evaluation of Outliers, Compositing and Variography

14.6.1 Evaluation of Outliers

Capping levels were determined using decile analysis and lognormal probability plots. Assays above the capping thresholds were viewed spatially; they are distributed throughout the higher grade portions of the deposit and no discreet high grade pods (above the capping thresholds) were observed.

Uncapped and capped assay gold grade statistics for fresh rock (refer to Table 14.5.4) and saprolite (Table 14.5.5) show that all domains, even before capping, have low variability (suggested by the low CVs) except for the 110 and 125 domains for saprolite.

Table 14.6.1 Mineralization Domain Codes

Mineralization Domain	Cap Level for Fresh Rock Au g/t	Cap Level for Saprolite Au g/t
101	1.5	1.0
105	2.8	2.1
110	9.0	5.0
125	30.0	10.0

14.6.2 Composites

A down-hole composite length of 2 m was chosen based on the proposed mining method, having a multiple of the average sample length, grade variability across the domains and mineralized domain thicknesses. New composites were started at tagged mineralization domain changes. Composite lengths vary a small amount to avoid small “residual” composites at the end of intervals. Assay grades were capped prior to compositing. The Fekola resource estimate is informed by a total of 24,101 composites. Statistics on capped 2 m gold grade composites by mineralization domain are shown on Table 14.6.2.

Table 14.6.2 Capped 2 m Gold Grade Composite Statistics by Mineralization Domain

Item	ALL	101	105	110	125
Number	24,101	12,255	5,133	4,590	2,123
Mean	1.063	0.214	0.662	1.643	5.675
Std Dev	1.96	0.14	0.29	0.81	3.93
Variance	3.85	0.02	0.08	0.66	15.41
CV	1.85	0.67	0.44	0.49	0.69
Maximum	30	1.195	2.028	8.589	30
75%	1.088	0.287	0.838	1.966	6.827
50%	0.41	0.18	0.648	1.503	4.331
25%	0.172	0.111	0.468	1.127	3.278
Minimum	0.005	0.005	0.005	0.031	0.287

14.6.3 Variography

Variograms (correlograms) were run (on 2 m capped composites) and modelled for each mineralization domain separately and all domains combined to evaluate gold grade spatial continuity and trends. The variogram models are listed on Tables 14.6.3 through 14.6.7. For all domains combined, domain 105 (0.5 to 1.0 g/t), and domain 110 (1.0 to 2.5 g/t) the direction of greatest continuity is at azimuth 347 to 351°, with a shallow dip ranging from -10° to -14°. The variogram ranges are 101 m, 181 m, and 205 m for all domains, domain 105 and domain 110, respectively. Domain 101 (0.1 to 0.5 g/t) and domain 125 (>2.5 g/t) do not show a strong down plunge anisotropy. Domain 101 typically occurs as a narrow low grade halo surrounding the higher grade domains. Domain 125 is more podlike than domains 110 and 105; additionally splitting the down-plunge shoot from the hanging wall areas may give better variograms.

Table 14.6.3 Variogram Models, All Mineralization Domains

All Domains					Structure 1		Structure 2			
Assay	**Domain**	**Direction**	**Orientation**	**Nugget**	**Sill**	**Range**	**Sill**	**Range**	**Type 1**	**Type 2**
				C0	C1	A1	C2	A2		
AUC	All	Downhole	00-->000	0.13	0.5	10	0.37	38	Spherical	Spherical
AUC	All	1	-15-->347	0.13	0.55	32	0.32	101	Spherical	Spherical
AUC	All	2	-72-->203	0.13	0.55	26	0.32	27	Spherical	Spherical
AUC	All	3	10-->260	0.13	0.55	22	0.32	39	Spherical	Spherical

Table 14.6.4 Variogram Models, Domain 101

101 Domain, 0.1 to 0.5g/t					Structure 1		Structure 2			
Assay	**Domain**	**Direction**	**Orientation**	**Nugget**	**Sill**	**Range**	**Sill**	**Range**	**Type 1**	**Type 2**
				C0	C1	A1	C2	A2		
AUC	101	Downhole	00-->000	0.38	0.34	6	0.28	15	Spherical	Spherical
AUC	101	1	-19-->343	0.38	0.53	23	0.09	51	Spherical	Spherical
AUC	101	2	-62-->213	0.38	0.53	19	0.09	64	Spherical	Spherical
AUC	101	3	20-->260	0.38	0.53	10	0.09	46	Spherical	Spherical

Table 14.6.5 Variogram Models, Domain 105

105 Domain, 0.5 to 1.0 g/t					Structure 1		Structure 2			
Assay	Domain	Direction	Orientation	Nugget	Sill	Range	Sill	Range	Type 1	Type 2
				C0	C1	A1	C2	A2		
AUC	105	Downhole	00-->000	0.38	0.23	4	0.39	13	Spherical	Spherical
AUC	105	1	-14-->351	0.38	0.45	55	0.17	181	Spherical	Spherical
AUC	105	2	-69-->219	0.38	0.45	48	0.17	113	Spherical	Spherical
AUC	105	3	15-->265	0.38	0.45	51	0.17	65	Spherical	Spherical

Table 14.6.6 Variogram Models, Domain 110

110 Domain, 1.0 to 2.5g/t					Structure 1		Structure 2			
Assay	Domain	Direction	Orientation	Nugget	Sill	Range	Sill	Range	Type 1	Type 2
				C0	C1	A1	C2	A2		
AUC	110	Downhole	00-->000	0.32	0.19	7	0.49	18	Spherical	Spherical
AUC	110	1	-10-->348	0.32	0.38	45	0.3	205	Spherical	Spherical
AUC	110	2	-76-->215	0.32	0.38	37	0.3	109	Spherical	Spherical
AUC	110	3	10-->260	0.32	0.38	24	0.3	56	Spherical	Spherical

Table 14.6.7 Variogram Models, Domain 125

125 Domain, > 2.5g/t					Structure 1		Structure 2			
Assay	**Domain**	**Direction**	**Orientation**	**Nugget**	**Sill**	**Range**	**Sill**	**Range**	**Type 1**	**Type 2**
				C0	C1	A1	C2	A2		
AUC	125	Downhole	00-->000	0.2	0.2	5	0.6	6	Spherical	Spherical
AUC	125	1	-10-->337	0.2	0.53	39	0.27	47	Spherical	Spherical
AUC	125	2	-72-->216	0.2	0.53	41	0.27	49	Spherical	Spherical
AUC	125	3	15-->250	0.2	0.53	23	0.27	45	Spherical	Spherical

14.7 Resource Block Model

14.7.1 Block Model Dimensions

A Datamine sub-celled block model was used as the basis for resource estimation. This model was re-blocked to a whole block of 5 x 10 x 5 m for resource reporting. The model was re-blocked to 5 x 20 x 10 m and converted to a percent model for mine planning work. Both the 5 x 10 x 5 m model used for resource reporting and the 5 x 20 x 10 m used for mine planning are regularized to the whole block size.

Block model specifications are listed in Table 14.7.1 and block model fields are on Table 14.7.2.

Table 14.7.1 Block Model Specifications

	Block Size			Model Origin			Number Of Blocks		
	X	Y	Z	X	Y	Z	X	Y	Z
Parent Cell Size-for grade estimation	5	20	10	241570	1385600	-380	222	145	57
Smallest Sub Cell Size	2.5	5	2.5	--	--	--	--	--	--
Reported Resource Block Size	5	10	5	241570	1385600	-380	222	290	114

Table 14.7.2 Block Model Fields

Block Model Export Fields	Description
IJK	Datamine IJK
XC	X center
YC	Y center
ZC	Z center
XINC	X size
YINC	Y size
ZINC	Z size
FILLVOL	Volume of block below Topo Surface
OBJ_INT	Dominant Code Mineralization Domain (100, 101, 105, 110, 125)
AUCOK	Capped Gold Grade – Ordinary Kriged, grade field used for resource reporting and mine planning.
RESCAT	Resource class: 2=Indicated, 3=inferred, 4=Exploration Potential (Not Reported)
SAP	Percentage of block in Saprolite
FRESH	Percentage of block in Fresh
OVB	Percentage of block in Overburden
MINZN	Percentage of block inside Mineralization domain
HGSH	High Grade Shoot (1=Inside High Grade Shoot, 0=Outside High grade Shoot)
SG	Weighted SG for block
PROCK	Percentage of block below Topo Surface
UDAUC	Block Grade within Mineral Domains (>0.1 grade shell)
WCODE	Waste Model Code: 5000=Overburden, 4000=SAP, 3000=Fresh
XMORIG	Model X Origin (No need to import)

Block Model Export Fields	Description
YMORIG	Model Y Origin (No need to import)
ZMORIG	Model Z Origin (No need to import)
NX	Numbers of Cells in X (No need to import)
NY	Numbers of Cells in Y (No need to import)
NZ	Numbers of Cells in Z (No need to import)

14.7.2 Gold Grade Estimation

Mineralization domain wireframes were coded to the blocks using sub-cells (minimum 2.5 x 5 x 2.5 m); the coded blocks were used as grade estimation domains. Gold grades were estimated into parent blocks (5 x 20 x 10 m) with Ordinary Kriging (OK) from 2 m capped composites for each domain. Dynamic anisotropic searching was used. This method of searching changes the orientation of the search ellipse during estimation to match the orientation of the triangulations of the solid model. Simplified overall orientation zones were used for this. The sub-cell model was re-blocked to 5 x 10 x 5 m whole blocks to account for the gradational change in gold grades seen at the domain contacts. Composites were shared across the saprolite / fresh boundary for estimation. In areas where saprolite is mineralized the mineralized portion looks similar in the saprolite relative to fresh. Grades were not estimated in overburden.

OK and Nearest Neighbour (NN) estimations were run, with mineral resources reported from the OK estimate. The estimation plan is presented in Table 14.7.3. Blocks that did not receive an estimated OK grade were assigned a gold grade of 0 g/t and a density value based on the weathering domain.

Table 14.7.3 Gold Grade Estimation Plan

Pass Number	Search Dimensions (m)			Max Composites Per Hole	Minimum Composites	Maximum Composites
	X	Y	Z			
Pass 1	30	60	50	4	5	16
Pass 2	45	90	75	4	5	16
Pass 3	75	150	125	4	4	12

14.7.3 Discussion of Re-blocking

Mineral Resources are reported from the re-blocked 5 x 10 x 5 m block model. The minimum sub-cell size of 2.5 m in the X and Z directions were re-blocked to 5m. The re-blocking accounts for the gradational change in grade observed over 2 to 3 m from the domain contacts as shown on contact plots (refer to Section 14.5.5) and on drillhole cross sections.

Mine planning was carried out on the re-blocked 5 x 20 x 10 m block model. The impact of re-blocking depends on cutoff grade. At 0.8 g/t Au cut-off, there is a 14% reduction in grade and 2% reduction in ounces when comparing the sub-celled model to the 5 x 20 x 10 m regularized block

model, within an optimized resource pit. At 1.1 g/t Au cutoff there is a 5% reduction in grade and 8% reduction in ounces.

14.7.4 Bulk Density

Average densities by modelled mineralization domains were applied to the block model and used in tonnage calculations. Densities used in the model are shown on Table 14.7.4.

Density measurements are done at site using the water submersion method (refer to Section 11.4, Density Analysis). An assumed density of 1.8 g/cm^3 was used for saprolite and overburden.

Table 14.7.4 Densities used in Tonnage Calculations

Mineralization Domain	Total Samples	% of Total	Avg. Au g/t	Avg Density g/cm^3
101 (0.1 to 0.5g/t)	527	48.0%	0.23	2.76
105 (0.5 to 1.0g/t)	264	24.1%	0.76	2.77
110 (1.0 to 2.5g/t)	187	17.1%	1.63	2.79
125 (>2.5g/t)	119	10.9%	6.80	2.83
Grand Total	**1,097**	**100.0%**	**1.31**	**2.77**

14.8 Model Validation

Block grade estimates categorized as Indicated and Inferred were validated using the following four methods:

- Visual comparison of block grades to composites on cross sections and levels.
- Comparison of global block statistics for the NN and OK Models.
- Swath plots to review potential local biases in the estimates.
- Comparison to previous models.

14.8.1 Visual Comparison of Composites to Block Grade Estimates

Block grade estimates were visually inspected relative to drillhole composite grades on sections and levels using paper plots and on screen. An example cross section through the block model is shown on Figure 14.8.1 and a level is shown on Figure 14.8.2.

The following observations are made:

- Mineralization domains coincide with the current structural and lithological understanding of the deposit as well as help control over projection of higher grade into areas with lower grade drilling.

- Local grade variability is sometimes high; however, block grade estimates reasonably represent composite grades.

- The “hanging wall” area has more grade variability than the main high grade shoot. It is possible that mining these areas will return higher tonnes and lower grade than estimated. More selective mining may be warranted in this area.

Figure 14.8.1 Cross Section Looking North, Block Model and Composite Gold Grades

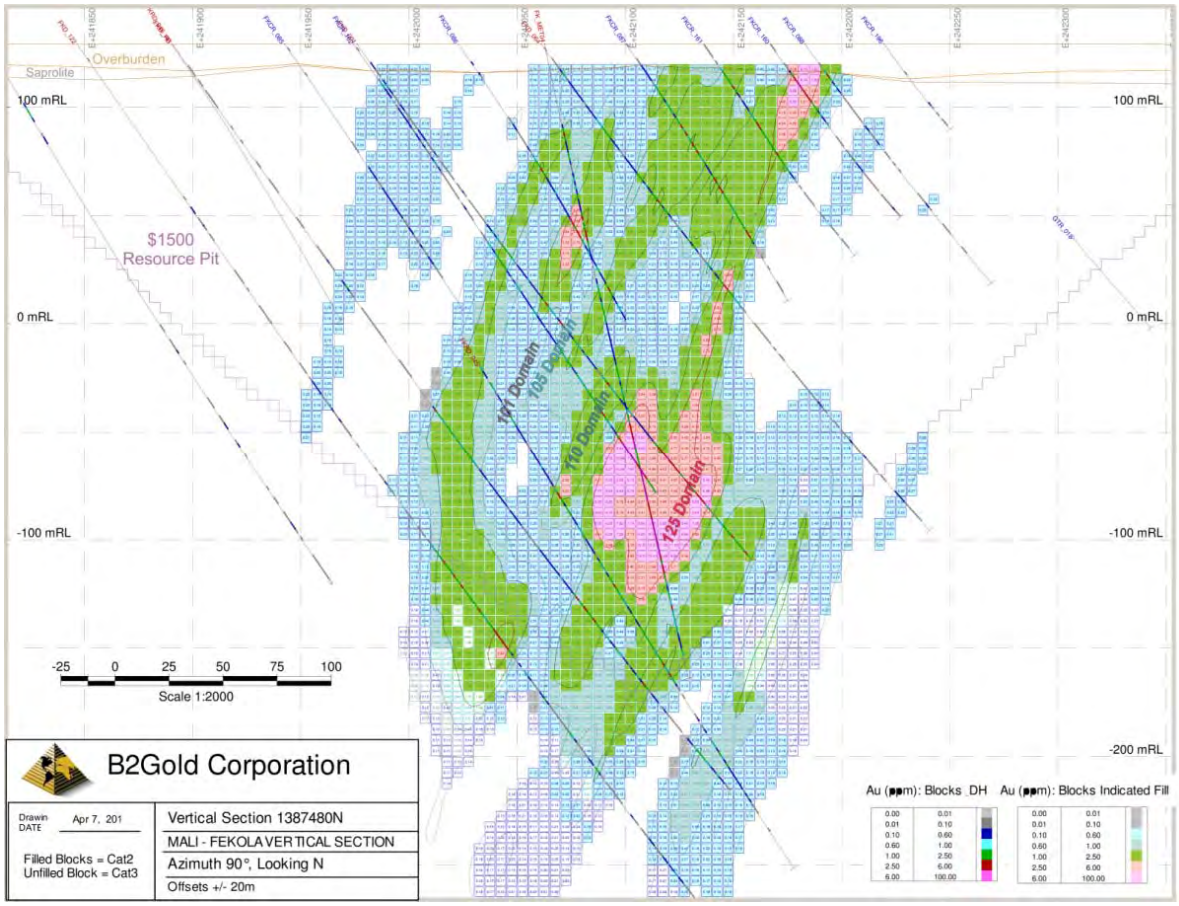
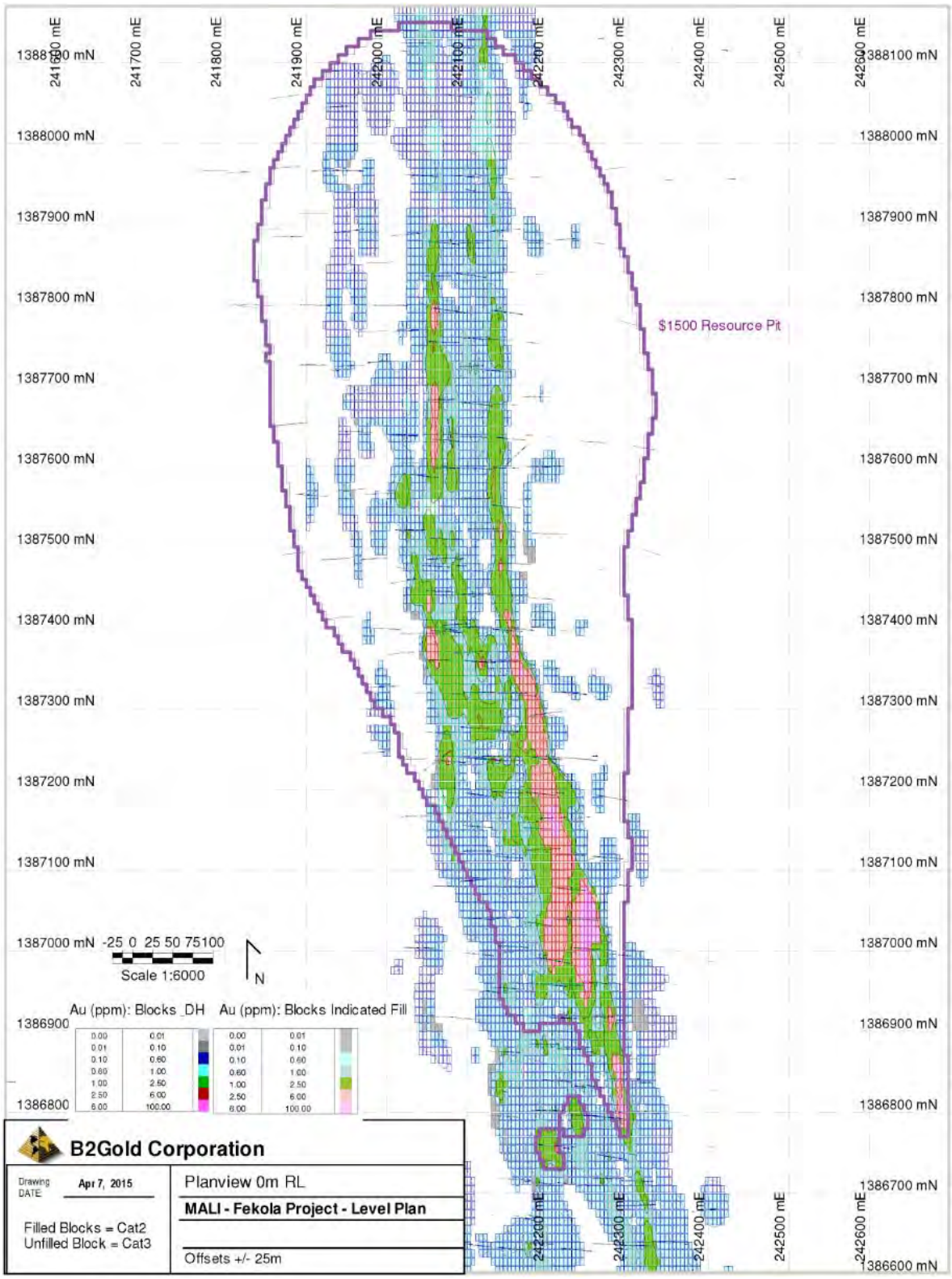


Figure 14.8.2 Plan View, Elevation 0 m, Block Model Gold Grades



14.8.2 Comparison Statistics

The global means at 0 g/t gold cut-off (Table 14.8.1) for OK and NN sub-cell models for individual mineralized domains and for all mineralized domains compare within acceptable levels for blocks categorized as Indicated plus Inferred.

Table 14.8.1 Comparison of Estimation Means at 0 g/t Cut-off

Indicated and Inferred Blocks by Domain	Au g/t OK Estimate (1)	Au g/t NN Estimate (2)	Percent Difference, OK relative to NN
101	0.202	0.208	-2.9
105	0.644	0.622	3.5
110	1.593	1.685	-5.5
125	5.583	5.761	-3.1
All	0.701	0.737	-4.9

(1) **OK= Ordinary Kriged Estimate, sub-cell model**
(2) **NN=Nearest Neighbour Estimate, sub-cell model**

14.8.3 Swath Plots

A check on local bias was done using swath plots showing (above 0 g/t cut-off) 'raw' 2 m composite grades, de-clustered composites using cell de-clustering and NN estimation, and OK grades from both the sub-celled and re-blocked resource-reporting grades. The 'swath plots' show mean grades by easting, northing and elevation through the Fekola resource model. The model grades for Indicated and Inferred blocks (not within the conceptual pit) were included in the comparison. A set was run for each mineralization domain and overall all domains. The graphs from the overall comparison are shown on Figures 14.8.3 to 14.8.5.

Coloured lines on the plots are described below:

- Yellow Line: Native Mean.
- Cyan Line: Cell de-clustered.
- Green Line: Re-blocked (5 x 10 x 5) OK – Resource Grade.
- Dark Blue Line: Sub-cell OK Grade (Prior to re-blocking).
- Red Line: NN Estimate – de-clustered.

Generally the OK curves are somewhat smoothed relative to the two declustered (NN and cell) distributions. The re-blocked distribution is slightly smoother than the sub-celled distribution, as expected. The OK estimates appear somewhat low in the lower grade domains when compared to cell and NN declustered means. Areas with large differences between the different estimations generally correspond to areas with a small number of composites, usually occurring at the flanks of the deposit where drill density is lower and blocks are more likely to be in the inferred category.

Figure 14.8.3 Comparison of Grade Estimates by Northing

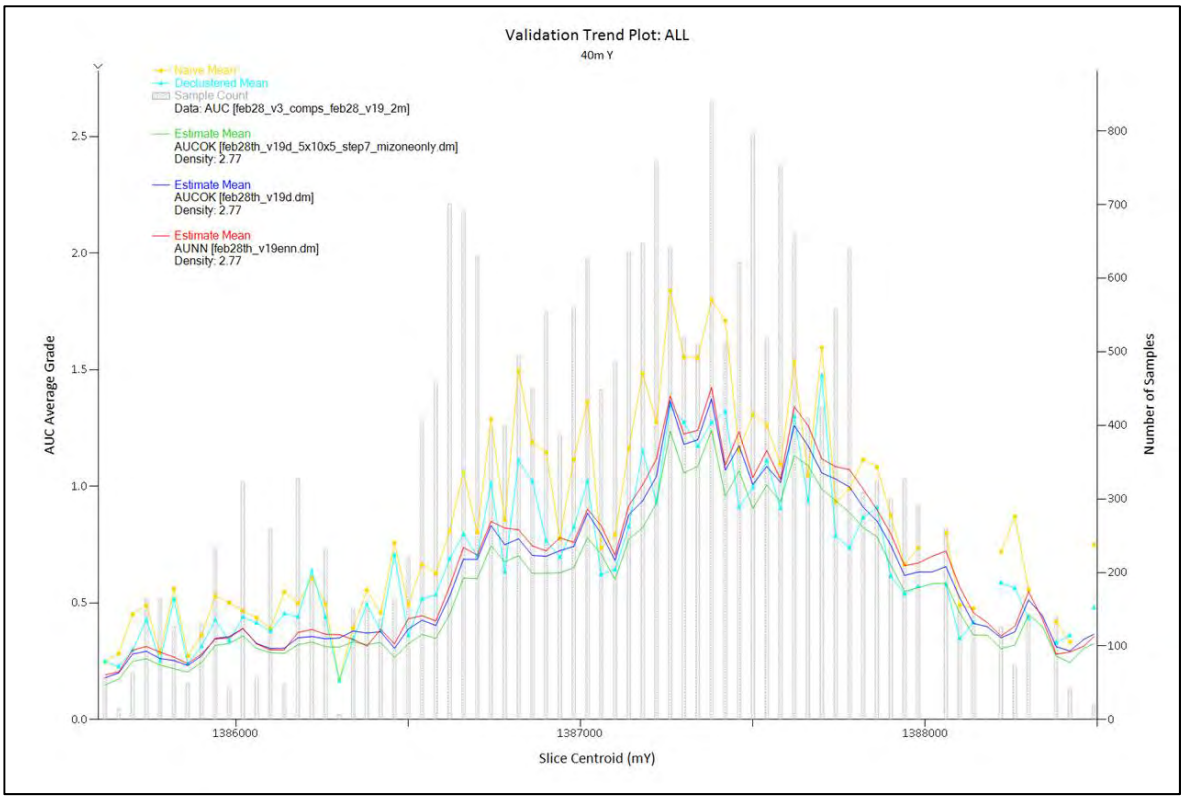


Figure 14.8.4 Comparison of Grade Estimates by Elevation

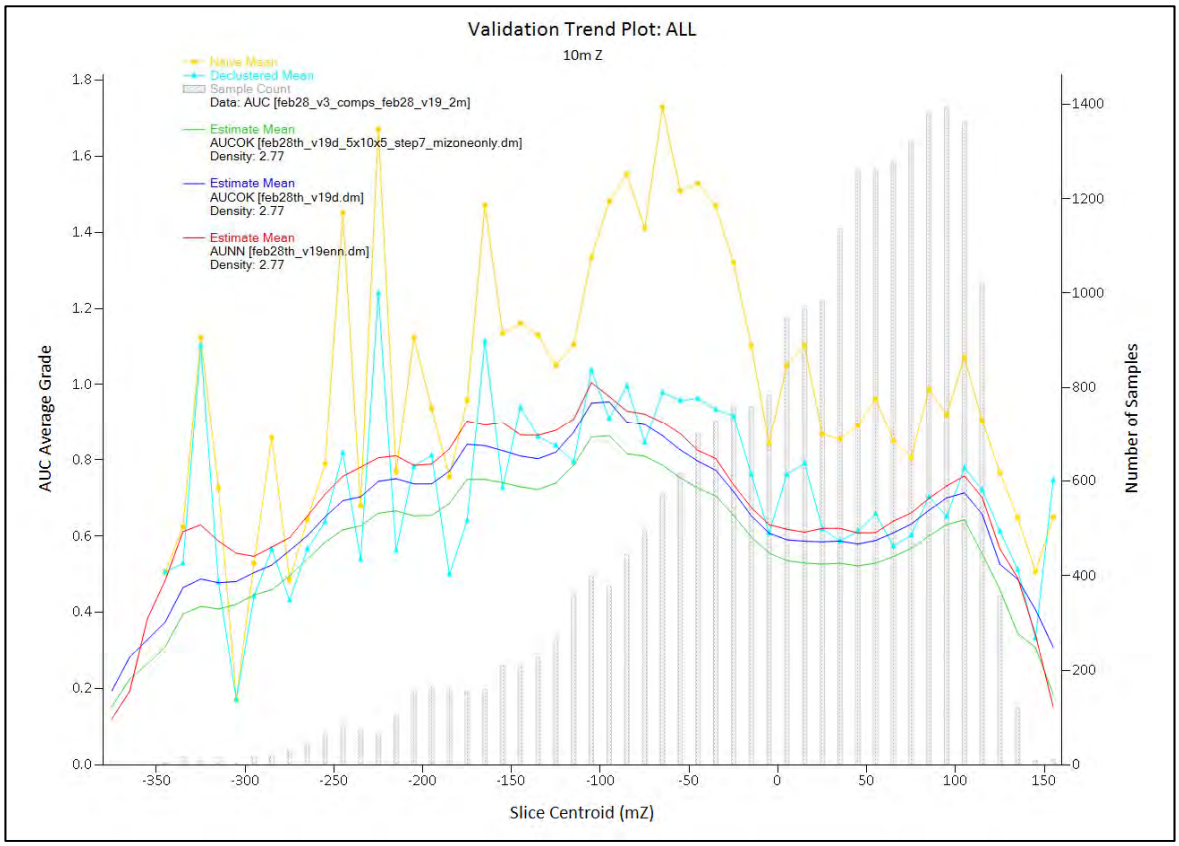
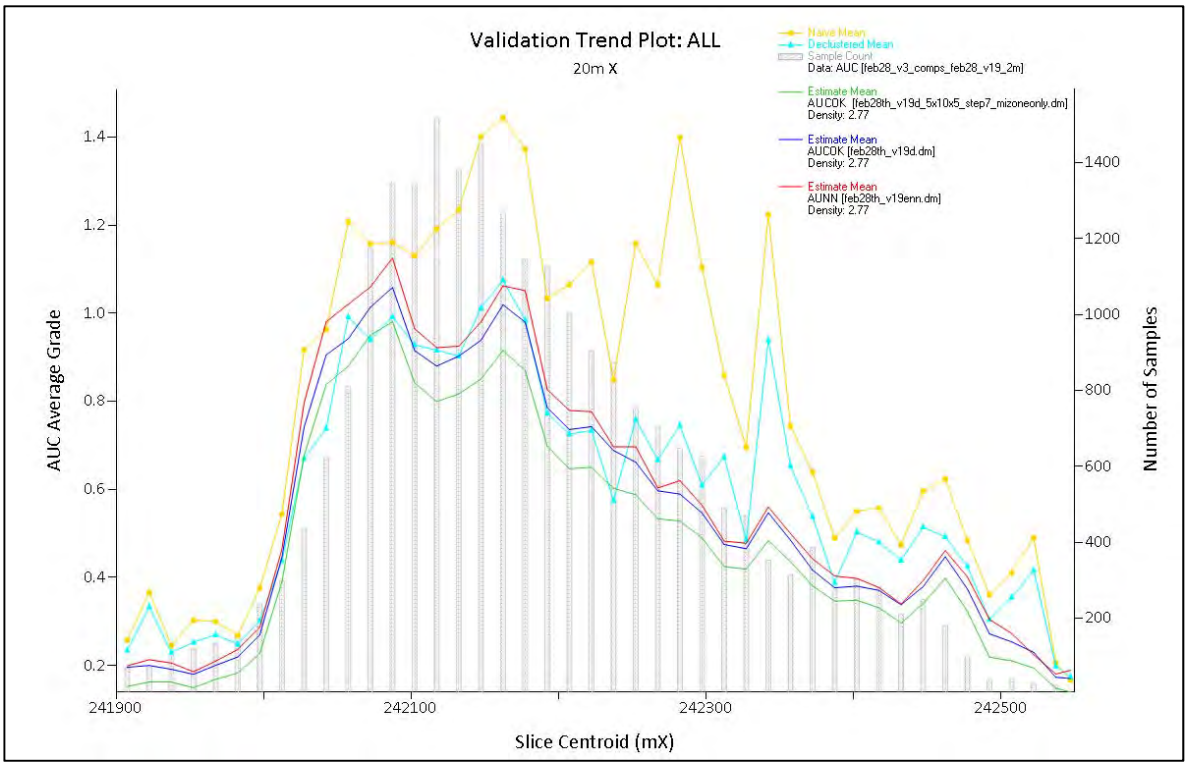


Figure 14.8.5 Comparison of Grade Estimates by Easting



14.8.4 Comparison to Previous Resource Models

Previous mineral resource estimates completed by MPR (January 2013 and August 2013, as reported in B2Gold, 2014) are multiple indicator kriged (MIK) models that are solely based on distributions of grade. The approach used to build B2Gold's February 2015 block model incorporates the current geological understanding of the deposit. Other important differences between the two models include no Measured Resources and a reduction in the outer limits of Indicated and Inferred resources in the February 2015 relative to the previous block models. Table 14.8.2 shows a summary comparison of salient differences in modelling methodology. Table 14.8.3 shows a comparison of the previous mineral resource results to the current one, at selected cut-off grades. At a gold cut-off grade of 0.6 g/t, the new model has 16.4% fewer ounces in Indicated compared to Measured plus Indicated Resources in the previous model, and a 31.4% increase in Inferred ounces relative to the previous model. The change in resource classification criteria as applied in the new model highlights areas with limited drill information. Additional drilling is planned with the goal to convert Inferred Resources in critical areas to Indicated Resources.

Table 14.8.2 Comparison of Modelling Methodologies, MPR August 2013 and B2Gold's February 2015 Model

	MPR August 2013 Model	B2Gold February 2015 Model
Drilling Available	439 RC (64,663m) 54 RC-DD (17,827m) 150 DD (44,659m) Total 643 drillholes (127,148 m)	440 RC (56,583m), 42 RC-DD (13,130m), 217 DD (53,915m) Total 699 drillholes (123,628m)
Interpretations- Weathered - Fresh	Interpretations on 40m spaced sections used to build surface from logged data.	Logged data used to build surfaces using Leapfrog software.
Interpretations- Mineralization Domains	0.1 g/t gold grade shell striking N to NNW, limited on east by Fekola Fault.	Mineralization domains at 0.1, 0.5, 1.0 and 2.5 g/t controlled by zones of high pyrite, folding and faulting
Panel or Parent Block Size	20x40x5m	5x20x10m, with subcells at Mineralization domain boundaries.
Estimate Reported to Block Size	5x10x5m	5x10x5m
Estimation Method	MIK	Domain controlled Ordinary kriging
Resource Classification- Drill spacing: Measured Indicated Inferred	 Up to 40x40m Up to 60x80m Greater than 80x160m	 No Measured 40x40m 80x80m
Mineral Resource Limits	Pit run using: - PEA[(1)] costs and metallurgical recovery - US$ 1,550/oz gold Reported above 0.6g/t cut-off	Pit run using: - Feasibility study preliminary costs, 92.4% recovery [(1)] - US$ 1,500/oz gold Reported above 0.6g/t cut-off

[(1)] **PEA=Preliminary Economic Assessment (B2Gold, 2014)**

Table 14.8.3 Comparison of B2Gold March 2015 Model to MPR August 2013 Model

Au Cutoff Grade (g/t)	B2Gold March 2015 Model						MPR August 2013 Model								
	Indicated			Inferred			Measured			Indicated			Inferred		
	Tonnes x 1000	Au g/t	Oz x 1000	Tonnes x 1000	Au g/t	Oz x 1000	Tonnes x 1000	Au g/t	Oz x 1000	Tonnes x 1000	Au g/t	Oz x 1000	Tonnes x 1000	Au g/t	Oz x 1000
0.5	67,293	2.03	4,383	9,990	1.58	507	65,700	1.75	3,690	27,200	1.81	1,580	7,800	1.55	380
0.6	61,574	2.16	4,281	9,055	1.68	490	59,000	1.89	3,580	24,700	1.94	1,540	6,900	1.69	370
0.7	55,202	2.34	4,148	7,572	1.89	459	52,900	2.03	3,450	22,400	2.07	1,490	6,100	1.82	350
0.8	50,412	2.49	4,033	6,429	2.09	432									
0.9	47,505	2.59	3,954	6,019	2.17	421									
1.0	44,964	2.68	3,877	5,704	2.24	411									
1.1	42,452	2.78	3,792	5,385	2.31	400									
1.2	39,875	2.88	3,697	5,014	2.40	387									

14.9 Mineral Resource Classification

Resource classification was assigned to parent blocks based on the following:

- Measured: no blocks assigned as Measured.
- Indicated: target 40 x 40 m drill spacing, includes blocks with an estimated grade using a minimum of two drillholes within 50 m and a minimum of one drillhole within 30 m.
- Inferred: target 80 x 80 m drill spacing, includes blocks with an estimated grade using a minimum of two drillholes within 100 m and a minimum of one drillhole within 60 m.

14.9.1 Reasonable Prospects of Eventual Economic Extraction

Mineral resources are reported above a selected cut-off grade, and within a conceptual Lerchs–Grossmann (LG) pit shell. Parameters used for the Mineral Resource pit shell are based on preliminary work completed for the OFS (in March 2015), using a gold price of US$ 1,500/oz, and are summarized in Table 14.1.1.

The calculated cut-off grade is approximately 0.60 g/t gold, which is used for resource reporting.

Table 14.9.1 Parameters used for Resource Pit Optimization Runs

Category	Unit
Price of Gold	US$1,500/oz
Royalty	7.65%
Metallurgical Recovery	92.4%
Throughput Cost (1)	US$ 25.12/t ore
Mining Cost	US$ 2.90/t ore plus sinking cost of 0.0327 per 10 m.
Pit Slopes	~43º (fresh) 33º (SAP/OVB)

(1) **Includes: Process, G&A, Grade Control and Rehabilitation cost.**

14.10 Mineral Resource Statement

The Mineral Resource estimate which supports the OFS has an effective date of 24 January 2015.

The estimate was prepared in accordance with the definitions set out by the CIM Definition Standards for Mineral Resources and Mineral Reserves adopted by the CIM Council on 10 May 2014 (2014 CIM definitions) as incorporated by reference in NI 43-101.

Table 14.10.1 Fekola Mineral Resources, as at 24 January 2015 [1,2,3,4,5,6]

Cutoff Grade Au (g/t)	Indicated			Inferred		
	Tonnes x 1000	Au g/t	Ounces x 1000	Tonnes x 1000	Au g/t	Ounces x 1000
0.6	61,574	2.16	4,281	9,055	1.68	490

(1) **Mineral Resources are estimated using best practices as defined by the CIM and reporting of Mineral Resources is compliant and in accordance with the disclosure requirements of NI 43-101.**
(2) **The Qualified Person for the Mineral Resources is Tom Garagan, P. Geo.**
(3) **Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.**
(4) **Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an indicated or measured Mineral Resource as a result of continued exploration.**
(5) **Mineral Resources are inclusive of Mineral Reserves.**
(6) **Mineral Resource numbers have been rounded to reflect the accuracy of the estimate and numbers may not add due to rounding.**
(7) **Mineral Resources are reported above a 0.6 g/t gold grade cutoff and within a pit shell using a gold price of US$1500/ounce, 92.4% metallurgical recovery, and March 2015 costs.**
(8) **Mineral Resources are reported on a 100% ownership basis.**

14.10.2 Sensitivity of Mineral Resource to Cut-off Grade

Mineral Resources are reported as of 24 January 2015 from the February 2015 block model, within a conceptual pit shell and above a cut-off grade of 0.6 g/t gold cut-off. The Mineral Resource estimate is highlighted on Table 14.10.2. Additional cases shown on the table illustrate the sensitivity of the estimate to changes in gold cut-off grade.

Table 14.10.2 Sensitivity Analysis to Mineral Resources by Gold Cut-off Grades, as at 24 January 2015

Cutoff Grade Au (g/t)	Indicated			Inferred		
	Tonnes x 1000	Au g/t	Ounces x 1000	Tonnes x 1000	Au g/t	Ounces x 1000
0.5	67,293	2.03	4,383	9,990	1.58	507
0.6	61,574	2.16	4,281	9,055	1.68	490
0.7	55,202	2.34	4,148	7,572	1.89	459
0.8	50,412	2.49	4,033	6,429	2.09	432
0.9	47,505	2.59	3,954	6,019	2.17	421
1	44,964	2.68	3,877	5,704	2.24	411
1.1	42,452	2.78	3,792	5,385	2.31	400
1.2	39,875	2.88	3,697	5,014	2.40	387

14.11 Recommendations

The following recommendations are made with respect to the Mineral Resource estimate:

- Additional drilling is needed in the hanging wall zone to confirm mineralization controls, grade continuity and the extent of the high grade mineralization domains.
- Before production commences, closer spaced drilling (on the order of 20 x 20 m) is needed to assess close spaced variability to allow testing of appropriate grade control drill spacing.
- Other non-linear estimation techniques, such as conditional simulation should be run as a confirmation to the grade tonnage curve.

14.12 Comments on Section 14

The QPs are of the opinion that the Mineral Resources for the Project, which have been estimated using core and RC drill data, have been performed to industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards, 2014.

FEKOLA GOLD PROJECT

NI 43-101 - MINERAL RESERVE ESTIMATES

3174-STY-002

Table of Contents

			Page
15.0	**MINERAL RESERVE ESTIMATE**		**15.1**
	15.1	Summary of Mineral Reserve Estimate	15.1
	15.2	Basis of Estimate	15.1
	15.3	Pit Optimisation Key Assumptions	15.2
		15.3.1 Resource Model	15.2
		15.3.2 Dilution and Ore Recovery	15.3
		15.3.3 Geotechnical Considerations	15.3
		15.3.4 Base Mining Cost Estimate	15.4
		15.3.5 Throughput Costs	15.5
		15.3.6 Gold Price, Royalty and Discounting	15.6
		15.3.7 Process Recovery and Cutoff Grades	15.6
	15.4	Pit Optimization Results	15.7
		15.4.1 Methodology	15.7
		15.4.2 Whittle Results and Pit Shell Selection	15.7
		15.4.3 Pit Optimization Sensitivity Results	15.9
	15.5	Mine Design Criteria	15.12
		15.5.1 Pit Development Strategy	15.12
		15.5.2 Pit Design Criteria	15.12
		15.5.3 Road and Ramp Design Criteria	15.13
		15.5.4 Waste Storage Facility Design Criteria	15.13
	15.6	Pit Stage Designs	15.13
	15.7	Pit Stage Mining Inventory	15.18
		15.7.1 Cut-off Grades for Ore Classification	15.18
		15.7.2 Mining Inventory by Cut-off Grade and Pit Stages	15.18
	15.8	Waste Storage Facility Designs and Site Layout	15.19
	15.9	Risk Assessment	15.21

TABLES

Table 15.1.1	Mineral Reserve Estimate as of 31 May 2015	15.1
Table 15.3.1	Pit Optimisation Preliminary Operating Cost Estimates	15.5
Table 15.3.2	Process Recoveries and Cut-off Grade	15.7
Table 15.4.1	Base Case Pit Optimisation Results Summary	15.8
Table 15.4.2	Pit Optimisation Sensitivity Results Summary	15.10
Table 15.5.1	Pit Slope Design Parameters	15.12
Table 15.7.1	Mining Inventory Summary by Pit Stages	15.19
Table 15.9.1	Project Development Risk Assessment	15.21

FIGURES

Figure 15.3.1	Pit Slope Design Sectors	15.4
Figure 15.3.2	Gold Extraction v Head Grade (B2Gold, May 15)	15.6
Figure 15.4.1	Unit Production Cost per Ounce v Optimal Pit Resource Tonnage	15.9
Figure 15.4.2	Extent of Optimal Pit Shells by Gold Price	15.11
Figure 15.6.1	Ultimate Pit Design	15.14
Figure 15.6.2	Plan View of Pit Stage Designs	15.15
Figure 15.6.3	Perspective and Section View of Pit Stage Designs	15.16
Figure 15.6.4	Northern Expansion of the Pit Limits in Six Stages	15.17
Figure 15.8.1	Waste Storage Facility Designs and Site Layout	15.20

15.0 MINERAL RESERVE ESTIMATE

15.1 Summary of Mineral Reserve Estimate

The Mineral Reserve estimate is based on the February 2015 block model prepared by B2Gold, as discussed in Section 14.

The Probable Mineral Reserve estimate, presented in Table 15.1.1, is based on the Indicated resources that fall within the final pit design limits.

Table 15.1.1 Mineral Reserve Estimate as of 31 May 2015

Classification	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (Moz)
Probable Reserves	49.17	2.35	3.72

1. **CIM definitions were used for Mineral Reserves.**
2. **The Qualified Person for Mineral Reserves is Peter Montano, P.E.**
3. **All Mineral Reserves were classified as Probable, based on the Indicated Mineral Resources.**
4. **The Mineral Reserve cutoff grade is 0.8 g/t Au based on a US$1,300/oz gold price.**
5. **The average metallurgical recovery is 92.8%, and the waste to ore ratio is 4.5:1 for Mineral Reserves.**
6. **The block model was internally diluted at the SMU size of 5 m E x 20 m N x 10 m RL. In addition, a waste dilution factor of 5% was applied in conversion to the Mineral Reserves to account for mining factors.**
7. **Sum of individual amounts may not equal due to rounding.**

The Qualified Person as defined in the CIM Definition Standards and required under National Instrument 43-101 is Peter D. Montano, PE. Mr. Montano is a registered engineer in the United States of America, Colorado, and is an employee of B2Gold Corp.

15.2 Basis of Estimate

At the request of B2Gold, the mine design and mine planning for the OFS undertaken by Mining Solutions Consultancy Pty Ltd. The mining study and the resultant Mineral Reserve estimate relied on the information from the following sources:

- Mr. Montano visited the site twice, 4 to 8 August 2014 and 7 to 11 December 2014. During these visits Mr. Montano visited the pit area, viewed a variety of ore and waste core, and toured the site in general. Mr. Montano also met with the geology team at the Fadougou camp and with office personnel in Bamako.

- Geotechnical appraisal and pit slope recommendations were provided by George, Orr and Associates (GOA) for feasibility study purposes in June 2014.

- Hydrogeological investigation results including ground water modelling and pit dewatering estimates were provided by Knight Piesold Consulting (KP) in September 2014.

- Environmental and social impact assessment for the project was provided by Epoch Resources Pty Ltd (Epoch) as part of the PFS in April 2013. The assessment is currently under revision to incorporate the current OFS results.

- The February 2015 resource model update was provided by B2Gold in early March 2015, and compared to the previous (MPR, August 2013) resource estimate.

- General administration, processing costs and process recovery estimates were provided by B2Gold in consultation with the Lycopodium Minerals Pty Ltd (Lycopodium) in March 2015, and revised in May 2015.

- The mining productivity and cost estimates are based on the information received from major equipment suppliers in March 2015, which is complemented with the data available from similar projects and operations.

- Mining parameters and mining costs were collated and estimated by Mining Solution Consultancy Pty Ltd (MSC) in consultation with B2Gold based on the experience, site visit findings, and the information received from the sources above.

The major tasks completed in the mining study for reserve estimation include the definition and review of study parameters, pit limit optimisations, cut-off grade analysis, and mine design.

As reported in Section 16, the production scheduling, mining equipment selection, and estimation of mining costs in the OFS further confirmed the basis of the mine design and the resultant Mineral Reserve estimate.

15.3 Pit Optimisation Key Assumptions

15.3.1 Resource Model

The pit optimisations and reserve estimates are based on the February 2015 resource model, as discussed in Section 14. The resources in the model were classified into Indicated and Inferred categories in compliance and accordance with the disclosure requirements of NI 43-101 and the CIM Definition Standards. Also in accordance with the requirements, only the Indicated category resources were used in the mining study and resultant reserve estimates. The sensitivity of the open pits for the inclusion of the Inferred resources was investigated separately in pit optimisations.

The updated surfaces for the topography, weathering zones and block densities were provided by B2Gold. The surfaces were compared to the earlier data before using in pit optimisations and volume / tonnage calculations.

The February 2015 block model used in the study contained the average gold grade estimate, resource classification, weathering zone classification, and density values for a regular block size of 5mE x 20mN x 10mRL. The grade estimate for the whole blocks was expected to incorporate sufficient internal dilution in representing the selective mining unit (SMU) for open pit mining.

15.3.2 Dilution and Ore Recovery

As part of the block model review process, the grade-tonnage data reported within the pit limits was compared to the previous (MPR August 2013) resource estimate. The variation of the resource estimates generated by B2Gold at different block (SMU) sizes were compared to the continuity, extent, dip, and plunge of the ore zones for an assessment of the dilution and ore loss assumptions.

After the reviews, a waste dilution factor of 5% and ore recovery factor of 100% was applied globally to the block model used in the mining study. The mining cost estimates include the grade control drilling and sampling costs to achieve sufficient data resolution for the delineation of the ore outlines.

15.3.3 Geotechnical Considerations

The geotechnical appraisal and pit slope recommendations for the study were provided by George, Orr and Associates (GOA).

Ground conditions in the proposed Fekola staged pits have been interpreted from evaluations made on geotechnical borehole cores (including use of an optical and acoustic imaging televiewer device), unconfined compressive strength (UCS) tests carried out on representative core samples, and local structural geological conditions.

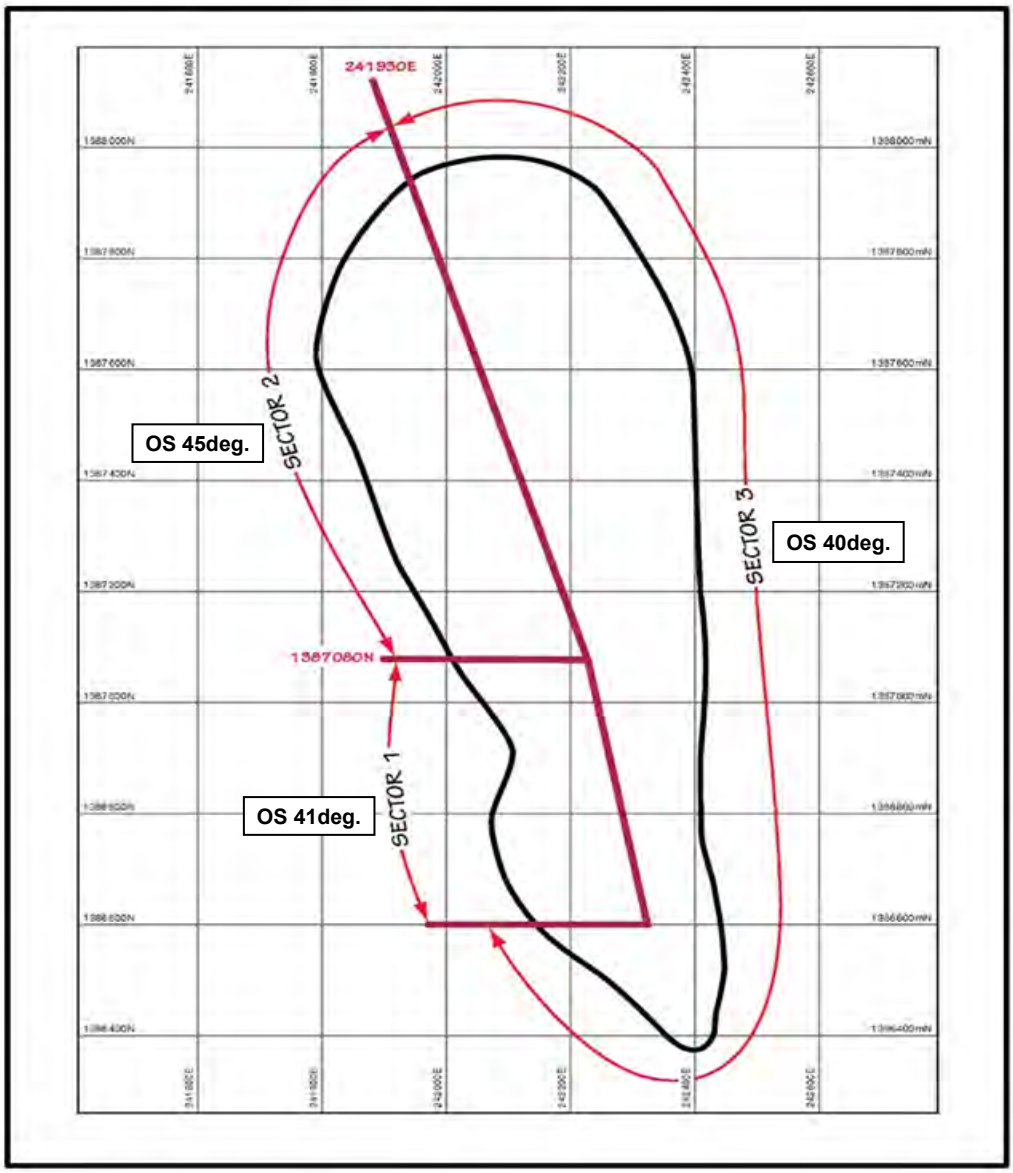
Future wall stability is expected to be governed principally by the presence, attitude, and shear strength parameters of the geological structures occurring within the walls. Limited analyses have been performed to assess the stability against large scale wall collapses (rotational wall failures).

Zones of highly fractured rock (termed "broken core zones") occur in the hanging- and foot-walls of the Fekola Fault. North-easterly striking faults are also inferred to occur at the deposit. Bedrock is covered by an approximately 10 to 15 m thick layer of transported (pebbly) alluvium.

The two dimensional stability analysis results imply that the future wall stability against potential rotational collapses affecting the integrity of the whole slope will remain adequate for mining purposes. This is provided that slopes are not mined at steeper overall angles than recommended and the effective wall depressurisation (i.e. dewatering, particularly in the east wall of the Stage 5 Pit) is carried out.

The pit wall slope design domains around the pit perimeter and the associated overall slope angles (OS) are shown in Figure 15.3.1. The overall slope angles vary from 40 to 45° around the pit rim, depending on mainly the extent and location of the broken core zones. The effect of access ramps on the pit walls were allowed in the definition of the overall slope angles.

Figure 15.3.1 Pit Slope Design Sectors



15.3.4 Base Mining Cost Estimate

The preliminary owner mining cost estimates were derived from the initial mining equipment productivity and cost estimates. The estimates were compared to the previous contract mining costs and the cost data for similar projects. The equipment ownership costs were included in the estimates for pit optimisation purposes, considering the relatively long mine life compared to the life cycle of the equipment.

The cost of mining the first bench at the surface is estimated at US$ 2.50/t mined. After adjusting the costs for blasting requirements, an incremental haulage cost of US$ 0.026/t per 10 m bench was applied to account for additional haulage costs as the pits deepen. The average mining cost

reported for the optimal pit shell is US$ 3.15/t mined, which includes the equipment ownership cost of US$ 0.35/t.

The OFS mining equipment and cost models were finalised after the preparation of the production schedule. The initial average mining cost estimate was revised from US$ 2.80/t to US$ 2.58/t in the final estimate. The capital cost for the mining equipment was reported separately in the final cost schedules.

15.3.5 Throughput Costs

The mining study was initially based on 5 Mtpa processing rate, which was revised to 4 Mtpa in May 2015 after the completion of the pit designs. The processing and general & administration costs for the 4 Mtpa base case in Table 15.3.1 were provided by B2Gold in consultation with Lycopodium.

The mining component of the throughput cost was revised from US$ 1.99/t ore in pit optimisation to US$ 2.24/t ore (+ 10%) in the final operating cost estimates.

Table 15.3.1 Pit Optimisation Preliminary Operating Cost Estimates

Category	Unit	Value	Source	Notes
Base Mining Cost	**US$ /t Mined**	**3.15**	MSC	Estimates by levels input to pit optimisation
Throughout Costs	**US$ /t Ore**	**27.04**		
Processing Cost	US$ /t Ore	21.00	B2Gold / Lycopodium	May 2015, revised from 5 Mtpa to 4 Mtpa
General and Administration	US$ /t Ore	4.05	B2Gold / Lycopodium	
Mine Throughput Costs	US$ /t Ore	1.99	MSC / B2Gold	
Mine Throughput Costs		1.99		
Mine Supervision	US$ /t Ore	0.51	MSC / B2Gold	Mine admin salaries and other costs
Ore Selective Mining	US$ /t Ore	0.20	MSC	Allowance for selective production
Grade Control	US$ /t Ore	0.41	MSC	Based on GC drilling and sampling estimate
Crusher Feed	US$ /t Ore	0.42	MSC / B2Gold	~50% ore handle and 50% direct tip ROM feed
Stockpile Rehandling	US$ /t Ore	0.30	MSC	Long-term stockpile rehandling (~30%)
Rehabilitation Allowance	US$ /t Ore	0.15	MSC	General allowance (waste dumps, tails, etc)

The ~10% decrease in the base mining cost and the ~10% increase in the mine throughput cost in the finalisation stage of the study has resulted in overall 4% lower operating cost per ounce of recovered gold in the final estimates.

15.3.6 Gold Price, Royalty and Discounting

A gold price of US$ 1,300/oz was used in the pit optimisations and the calculation of the break-even cut-off grade for reserves reporting. The pit optimisation sensitivities included the variation of ±15% (US$ 1,100/oz and US$ 1,500/oz) around the base case gold price.

The 7.65% royalty was deducted from the revenue by using the net gold price after royalty in pit optimisations. The operating cash flows were discounted at 5% per annum to calculate the indicative NPV values for the comparison of optimal pit shells and production schedule options.

15.3.7 Process Recovery and Cutoff Grades

The grade dependent equation used for the estimation of the process recoveries was derived from the test work results plotted in Figure 15.3.2. The 1% reduction is to scale the laboratory test results to the processing plant conditions in the recovery equation:

- Expected Gold Recovery (%) = 1.6705 ln(x) + 92.218 - 1.0

Figure 15.3.2 Gold Extraction v Head Grade (B2Gold, May 15)

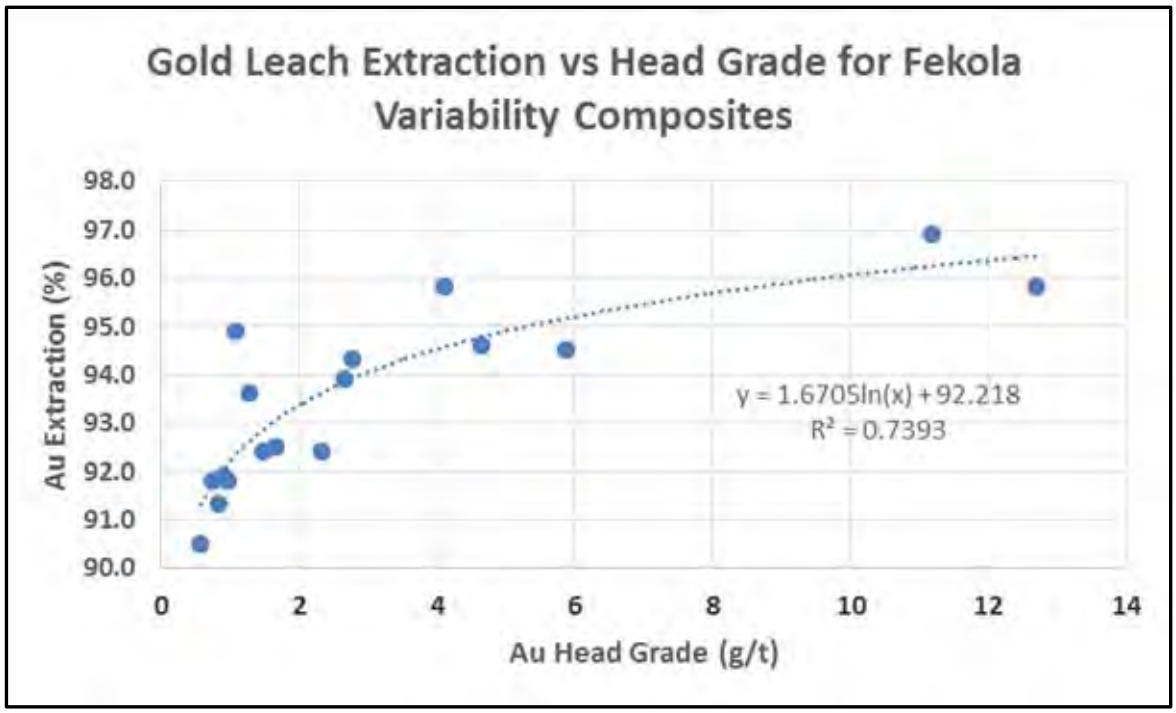


Table 15.3.2 below outlines the calculated gold recoveries for the life of mine average grade, the scheduled high and low grade mill feeds, and the reserve cut-off grade.

The block model cut-off grade of 0.80 g/t used in the reserve estimate is based on the 92.8% process recovery and the throughput cost, gold price, royalty, and dilution parameters reported in the sub-sections above.

Table 15.3.2 Process Recoveries and Cut-off Grade

Category	Unit	Value	Notes
		Recovery	Head (g/t)
Average LOM Recovery	%	92.8%	2.35 Average from production schedule
Recovery First 7 Years	%	93.1%	3.10 Rec% = 1.6705 ln(x) + 92.218 – 1
Low Grade Recovery	%	91.1%	0.95 Rec% = 1.6705 ln(x) + 92.218 – 1
Recovery at Cut-off Grade	%	90.8%	0.80 Rec% = 1.6705 ln(x) + 92.218 – 1
Cut-off Grade			
Break-Even COG	g/t Au	0.77	Cut-off grade after dilution
Resource Model Cut-off Grade	g/t Au	0.80	Applied to model for reserve estimates before dilution
Waste Distribution	%	5%	Mining factors in addition to model internal dilution

15.4 Pit Optimization Results

15.4.1 Methodology

Pit optimisations were carried out using Whittle Four-X pit optimisation software. For the given block model, cost, recovery and slope data, Whittle Four-X software determines a series of incremental pit shells, in which each shell is an optimum for a slightly higher price factor.

In the analysis of the incremental pit shells, indicative net present values (NPV) are calculated by discounting the preliminary cash flows over time. The reported NPVs in pit optimisation results are indicative operating values for relative comparison purposes only.

As well as the indicative NPVs, the incremental operating cost per ounce for the pit shells is also reported to guide the pit shell selection and design process.

In addition to the base case pit optimisation to determine the ultimate pit limits, further optimisations were carried out to determine the sensitivities around the base case results.

15.4.2 Whittle Results and Pit Shell Selection

Table 5.4.1 below summarises the quantities for the five pit shells selected from the base case optimisation. Pit shell 33 is identified as the optimal based on the maximum indicative NPV criterion. The final pit design quantities are compared to the optimal pit shell quantities in the second part of the table.

Table 15.4.1 Base Case Pit Optimisation Results Summary

Shell No.	Resources		Waste	Total	W/O Ratio	Recovered Ounces	Total Cost[1-3]	Operating Surplus	Unit Cost[1-3]	Indicative Op. NPV
	kt	Au g/t	t	kt	kt	koz	US$'000	US$'000	US$ /oz	US$'000
10	8,203	3.09	13,749	21,952	1.68	760	287,222	625,513	378	581,905
11	17,480	2.76	38,379	55,859	2.20	1,445	642,783	1,091,463	445	957,665
15	29,659	2.63	80,350	110,009	2.71	2,334	1,142,788	1,659,203	490	1,349,381
33	45,804	2.41	182,651	228,445	3.99	3,292	1,957,238	1,995,274	595	1,526,201
39	49,854	2.35	216,858	266,712	4.35	3,490	2,190,105	2,000,035	628	1,517,103
Pit Design	49,173	2.35	219,373	268,545	4.46	3,450	2,099,305	2,042,742	608	1,565,756
Pit 33%	107%	98%	120%	118%	112%	105%	107%	102%	102%	103%
Pit 39%	99%	100%	101%	101%	103%	99%	96%	102%	97%	103%

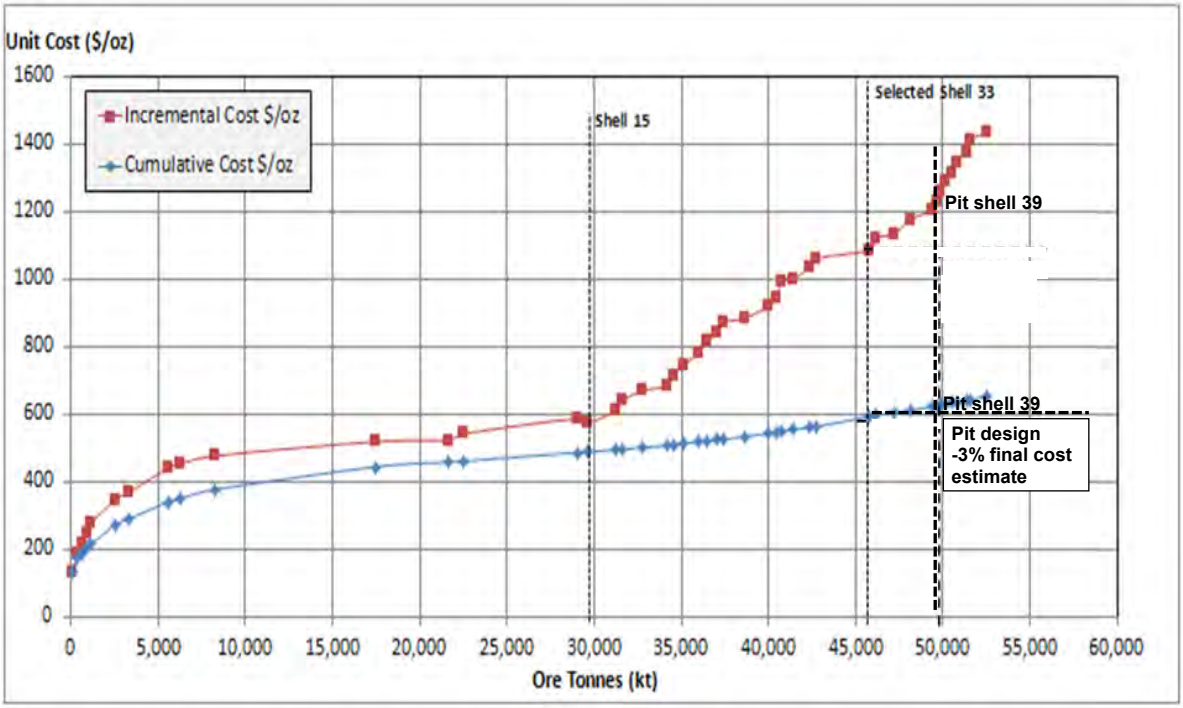
1. **The mining costs in pit optimisation reports include the ownership cost for mining equipment approximately.**
2. **The mining cost estimate for pit design is adjusted to include the equipment ownership cost on the same basis approximately.**
3. **The final (base + throughput) mining cost estimate is 7% lower than the preliminary cost in pit optimisation on the same $ /t basis.**

As seen in 15.4.1, the pit design is significantly (18%) larger than the optimal pit shell 33 due to the revision of the throughput from 5 Mtpa to 4 Mtpa in May 2015 after the completion of the mine design. However, the comparison of the values in Table 15.4.1 confirms the economic basis of the pit design as follows:

- The ore tonnage, total tonnage and the recovered ounces reported for the pit design are within 1% of the pit shell 39 quantities.
- The indicative NPVs do not change significantly for the expansion from pit shell 33 to pit shell 39.
- The operating NPV for the pit design is slightly higher (+3%) than the both pit shells 33 and 39.
- Based on the above points, pit shell 39 can be accepted as the optimal for the pit design rather than the initially selected pit shell 33.

For the base case pit shell sequence, the incremental and cumulative unit operating costs (US$/oz) are plotted against the optimal pit resource tonnage in Figure 15.4.2. The cost curves are relatively flat and the average cost is below US$ 500/oz up to 30 million tonnes of resource reported within the pit shell 15. This relatively high grade and low stripping ratio resource generates 88% of the operating NPV of the project. Hence, the staged pit development strategy improves the cash flow in the earlier years of the schedule and reduces the capital payback period significantly.

Figure 15.4.1 Unit Production Cost per Ounce v Optimal Pit Resource Tonnage



15.4.3 Pit Optimization Sensitivity Results

Table 15.4.2 below summarises the resultant optimal pit shell quantities for the studied sensitivity cases. The results are sorted from the worst to the best sensitivity case depending on the indicative NPV results. The percentage variation of the results around the base case is presented in the second part of the table.

As expected, the optimal pit size and operating value is most sensitive to a change in the gold price. A variation of ±15% in gold price causes a change of approximately ±30% in the operating cash flow, ±10% in the recovered ounces, and -25% and +15% in the total tonnes mined.

The optimal pit size is moderately sensitive to 15% increase in the throughput or mining costs, resulting in a reduction of approximately -20% in the optimal pit size and -5 to -10% in recovered ounces. The -5% decrease in the operating NPV is less significant.

Although the inclusion of the Inferred resources add approximately 18% to the optimal pit resource tonnes and 25% to the total tonnes mined, the additional resources (8 Mt) are relatively low grade (2.0 g/t) and high stripping ratio (6.6:1 t:t). The Inferred resources can potentially add 7% to the operating NPV of the Project.

Ramping up the throughput rate from 4 Mtpa to 5 Mtpa can potentially increase the pit size by 5%, mined resource tonnage by 15% and indicative operating NPV by 8% (US$ 130 million).

A ±3° change in the overall slope angles affects the operating NPV of the project within a ±4% range.

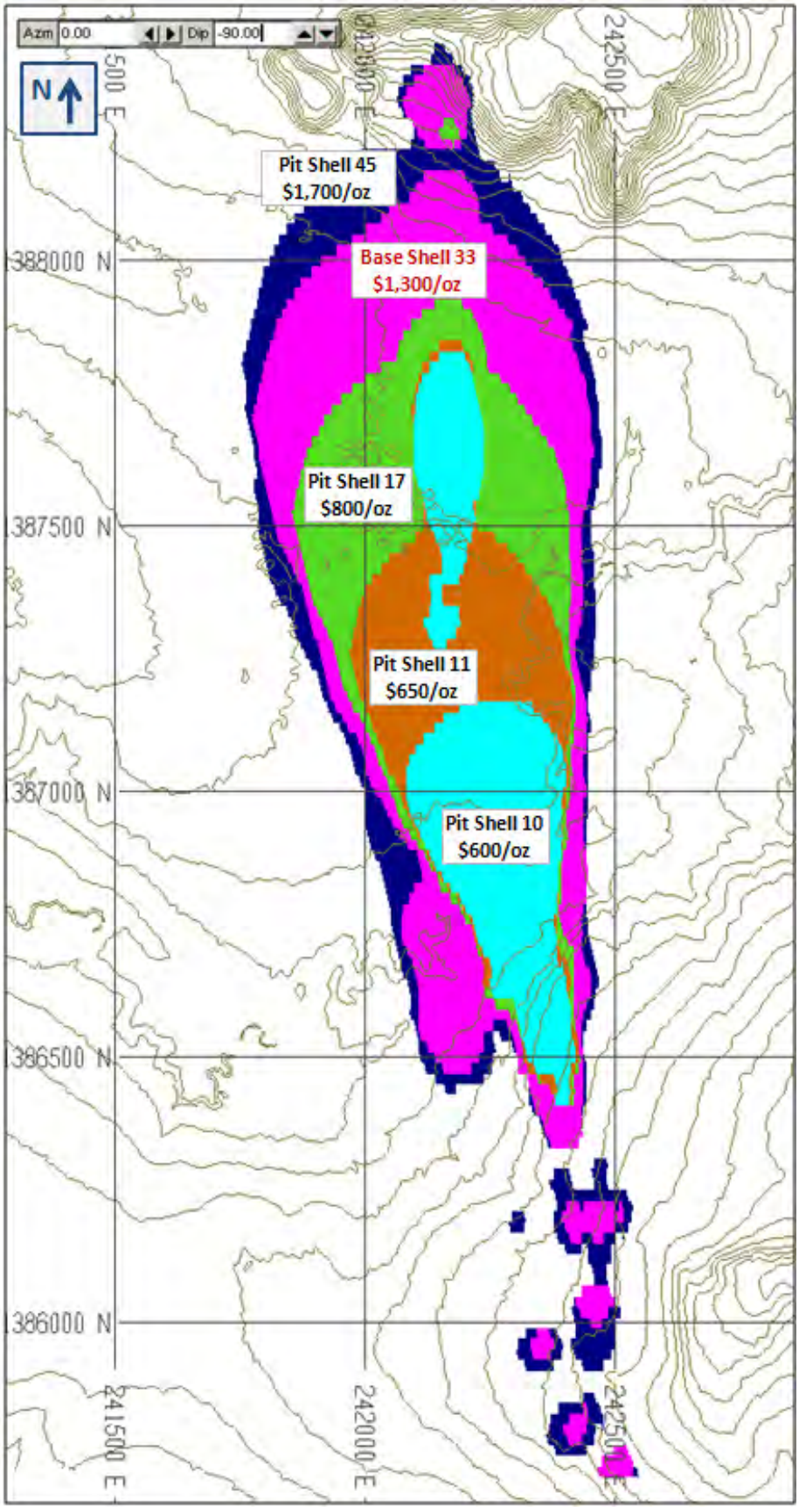
Table 15.4.2 Pit Optimisation Sensitivity Results Summary

Case	Pit No	Resource kT	Resource Au g/t	Waste kT	Total kT	W/O Ratio t/t	Recv'd Ounces koz	Operating Surplus $'000	Total Cost $/t Ore	Total Cost $/oz	Avg. NPV $'000
Optimal Pit Shell Quantities											
-15% Gold Price	28	37,102	2.63	137,605	174,707	3.71	2,912	1,408,939	41.8	532	1,119,903
+15% Throughput Cos	31	39,253	2.57	148,465	187,718	3.78	3,010	1,806,051	46.0	601	1,424,079
+15% Mining Cost	31	42,318	2.44	145,400	187,718	3.44	3,084	1,872,927	43.2	593	1,448,470
-3deg Slope	33	42,918	2.42	165,419	208,337	3.85	3,101	1,911,620	42.2	584	1,466,600
Base Case	33	45,804	2.41	182,651	228,455	3.99	3,292	1,995,274	42.7	595	1,526,201
+3deg Slope	33	46,273	2.42	171,166	217,439	3.70	3,336	2,068,088	41.9	581	1,576,508
-15% Mining Cost	35	47,217	2.39	194,088	241,305	4.11	3,363	2,104,277	40.9	575	1,599,699
Incl. Inferred	33	54,029	2.36	228,641	282,670	4.23	3,802	2,208,154	43.6	620	1,625,949
-15% Throughput Cost	35	52,155	2.23	189,150	241,305	3.63	3,465	2,201,577	37.6	565	1,636,623
5Mtpa Throughput	35	49,406	2.32	191,899	241,305	3.88	3,410	2,092,920	40.5	587	1,655,609
+15% Gold Price	38	54,419	2.21	211,286	265,705	3.88	3,583	2,653,228	42.5	645	1,958,460
As a Percentage of Base Case											
-15% Gold Price	28	81%	109%	75%	76%	93%	88%	71%	98%	89%	73%
+15% Throughput Cos	31	86%	107%	81%	82%	95%	91%	91%	108%	101%	93%
+15% Mining Cost	31	92%	101%	80%	82%	86%	94%	94%	101%	100%	95%
-3deg Slope	33	94%	100%	91%	91%	97%	94%	96%	99%	98%	96%
Base Case	33	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
+3deg Slope	33	101%	100%	94%	95%	93%	101%	104%	98%	98%	103%
-15% Mining Cost	35	103%	99%	106%	106%	103%	102%	105%	96%	97%	105%
Incl. Inferred	33	118%	98%	125%	124%	106%	115%	111%	102%	104%	107%
-15% Throughput Cost	35	114%	93%	104%	106%	91%	105%	110%	88%	95%	107%
5Mtpa Throughput	35	108%	96%	105%	106%	97%	104%	105%	95%	99%	108%
+15% Gold Price	38	119%	92%	116%	116%	97%	109%	133%	99%	108%	128%

The plan view of series of optimal pit shells in Figure 15.4.2 shows the extents of the ultimate pit limits as the gold price varies. The lowest cost pit development starts at the southern end of the deposit where the high grade core mineralisation is closest to the surface, then expands deeper to the north following the plunge of the high grade core.

The most sensitive areas for the variation of the study parameters are to the north where the stripping ratio is higher and at the southwest where the mineralisation outside the main orebody is patchy and low grade. In general, the pit limits to the west and east do not change significantly as the relatively well defined high grade mineralisation drives the pit floor in the optimisation process.

Figure 15.4.2 Extent of Optimal Pit Shells by Gold Price



15.5 Mine Design Criteria

15.5.1 Pit Development Strategy

The staged pit development strategy was the key in the production schedules to defer the waste mining requirements and bring forward the mining of high grade ore. The 320 m deep ultimate pit is planned for development in a sequence of seven 150 to 250 m wide and 500 to 750 m long stages (cutbacks).

The staged pit development will also mitigate the geological, geotechnical and economic risks for the project considering the 1.9 km length of the proposed Fekola open pit. The design of the future pit stages during the operations, especially the last two stages with higher production cost per ounce, can be adjusted progressively depending on the operational experience, exposed ground conditions and changes in economic conditions.

The cutbacks need to be accessed through temporary ramps in the initial stage of development from the surface. The majority of these temporary ramps will be mined after acting as catch (safety) berms between the successive cutbacks mined at different levels. The remaining ramps on the final pit walls will act as geotechnical berms (i.e. wider berms to limit the inter-ramp slope angle) to form a “stacked” slope design.

15.5.2 Pit Design Criteria

The definition of the pit slope domains and the recommended slope design parameters used in the pit optimisations and designs are detailed in Table 15.5.1. The slope design may vary slightly at the lower levels of the pit design to maximise the ore recovery where the access ramps can act as berms.

Table 15.5.1 Pit Slope Design Parameters

Rock Zone	Sectors	Depth	Bench Height (m)	Face Angle (deg)	Berm Width (m)	Inter-Berm Angle (deg)
Weathered Zone	All Sectors	Surf. to -20 m	10	55	8	~34
Fresh Broken Zone (BCZ)	Section 1 Sector 3	-60 m to -110 m -20 m to -320 m	10	70	8	~41
Fresh Rock (Outside NCZ)	All Remaining	All Remaining	20	70	8	~53

BCZ: Broken Core Zone

Sector 1: West wall between 1386600mN and 1387080mN
Sector 2: West wall to the north of Sector 1
Sector 3: West wall to the south of Sector 1 and all remaining walls

An average cutback width of 200 m was maintained in the design of the six consecutive cutbacks expanding the pit from south to the north. The size of the last cutback Stage 7 at the southwest is relatively small, but this will not be an issue operationally towards the end of the mine life as the processing of the low grade stockpiles will start.

A minimum mining width of 25 m (dual lane ramp width) was adopted for the floor of the ultimate pit design. The temporary floors of the pit stages were designed wider (~40 m) not to constrain the mining equipment unnecessarily as these floors would be mined in the subsequent pit stage.

15.5.3 Road and Ramp Design Criteria

A nominal ramp and road width of 25 m, including drainage and safety windrow, was used for dual lane truck operation in the mine design. Ramp widths were reduced to 15 m in mining the lower levels of the Stage 6 and 7 cutbacks developing the final pit walls. Passing bays were provided at approximately 200 m intervals for the single lane section of the ramps. A ramp gradient of up to 1 in 9 (11%) was used in the mine design for both single and dual lane ramps.

15.5.4 Waste Storage Facility Design Criteria

An overall slope angle of 18° was used in the design of the waste dump faces with the 5 m berms located at 10 m vertical intervals.

The temporary ramp exits from the pit stages were designed to distribute the waste across the waste storage areas and the TSF embankment. The size and height of the two waste dumps at the southwest and northeast were adjusted to minimise the haulage costs and leave the mineralisation corridor open to the south for potential development of shallow pits (or sterilisation) in the future.

15.6 Pit Stage Designs

The Fekola ultimate pit design shown in Figure 15.6.1 is maximum 1.9 km long, 750 m wide and 320 m deep. The plan view of the seven cutbacks designed to for the development of the ultimate pit limits is shown in Figure 15.6.2. The perspective and section views of the pit stages can be seen in Figure 15.6.3.

The Stage 1 pit design shown in Figure 15.6.4 is 400 m wide, 800 m long and 160 m deep, relatively large compared to the subsequent cutbacks to sustain the required 4 Mtpa high grade ore production rate at the start of the operations. The starter pit targets the relatively higher grade and shallower mineralisation at the south end of the deposit.

The two ramps developed in Stage 1 pit from the east for accessing the ROM pad and from the west for accessing the waste storage facility will join at approximately 50 m depth. The established ramp system will continue further north to serve the deeper levels of the subsequent cutbacks as the pit expands in this direction.

Figure 15.6.1 Ultimate Pit Design

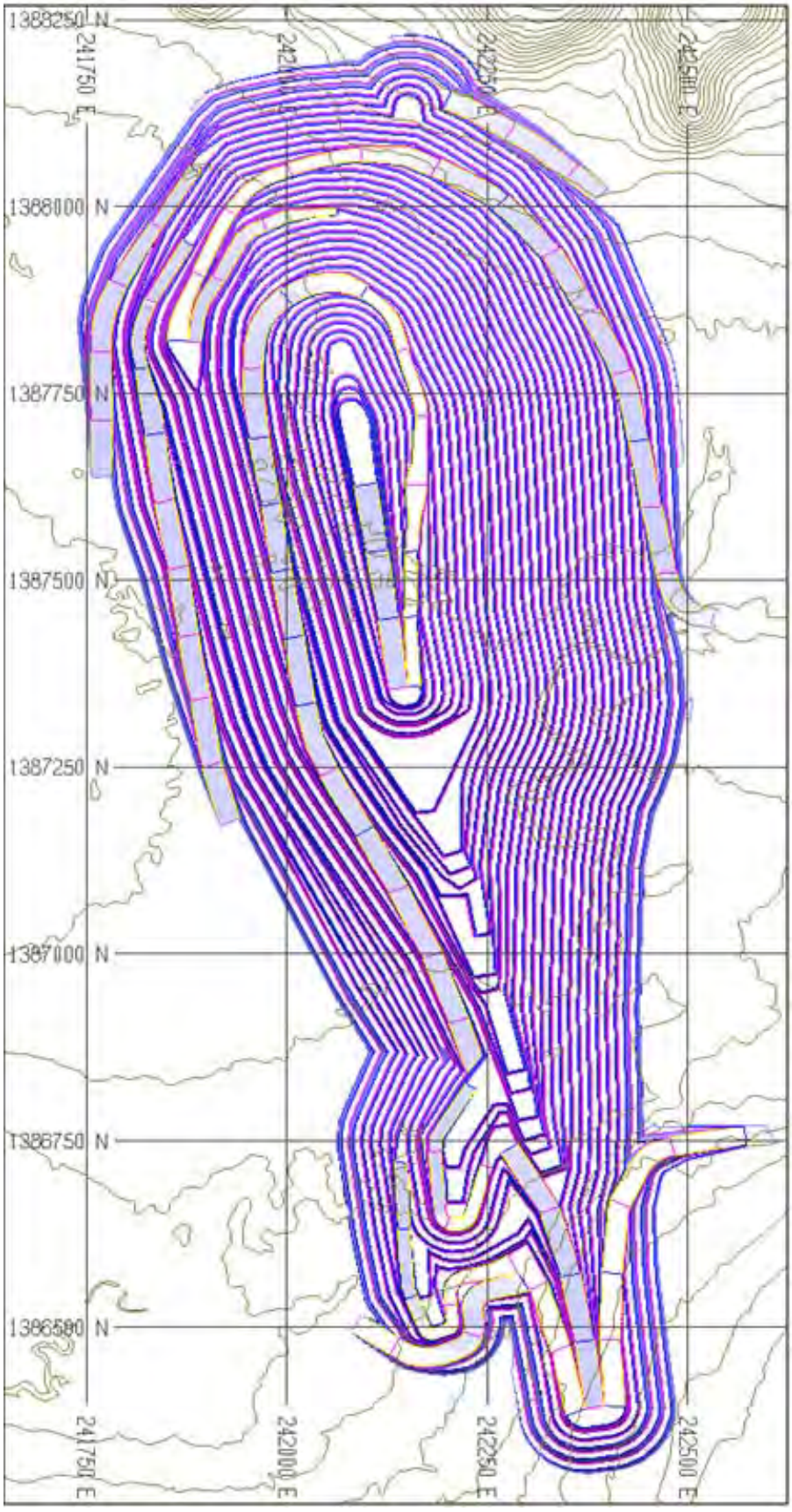


Figure 15.6.2 Plan View of Pit Stage Designs

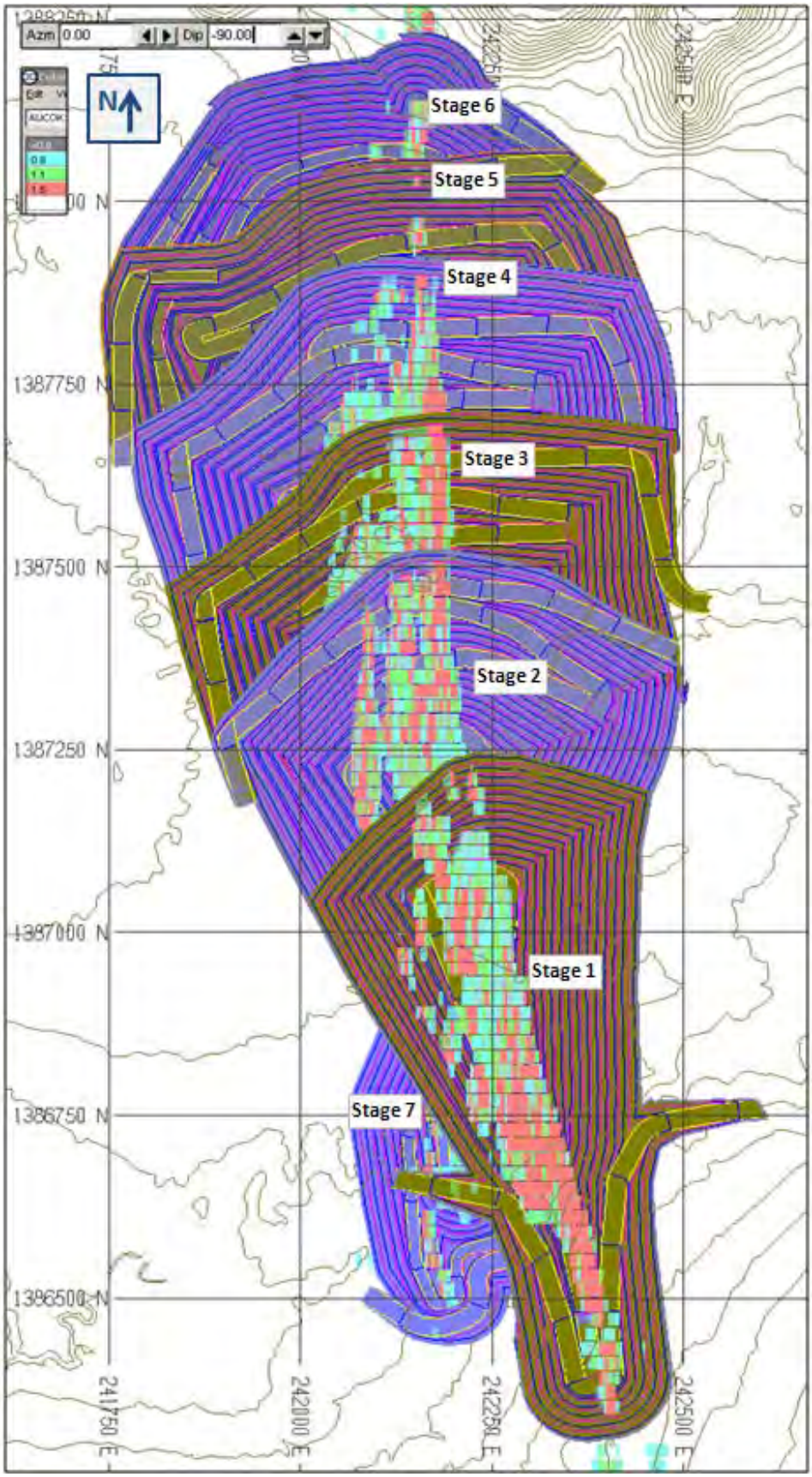
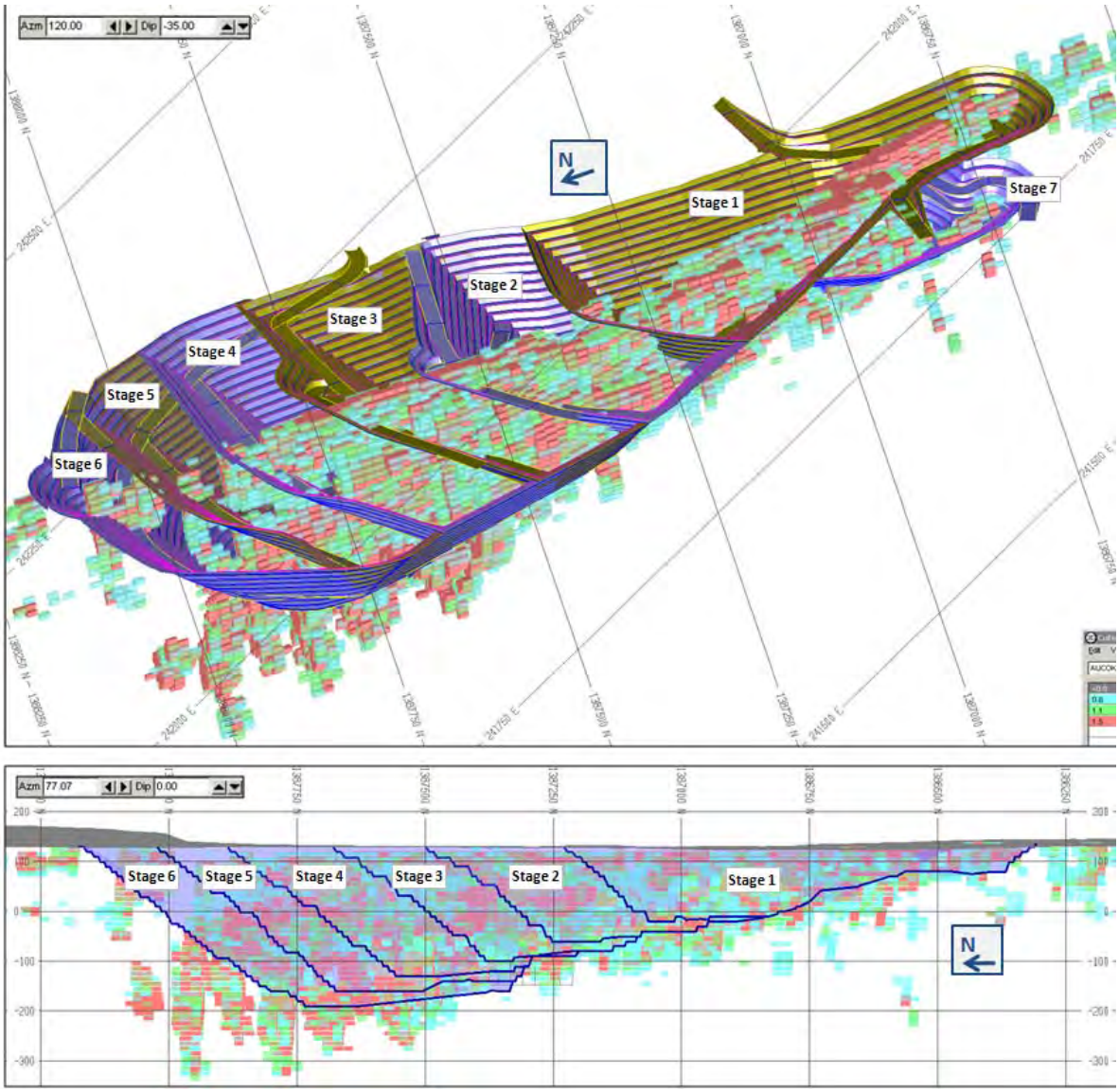


Figure 15.6.3 Perspective and Section View of Pit Stage Designs

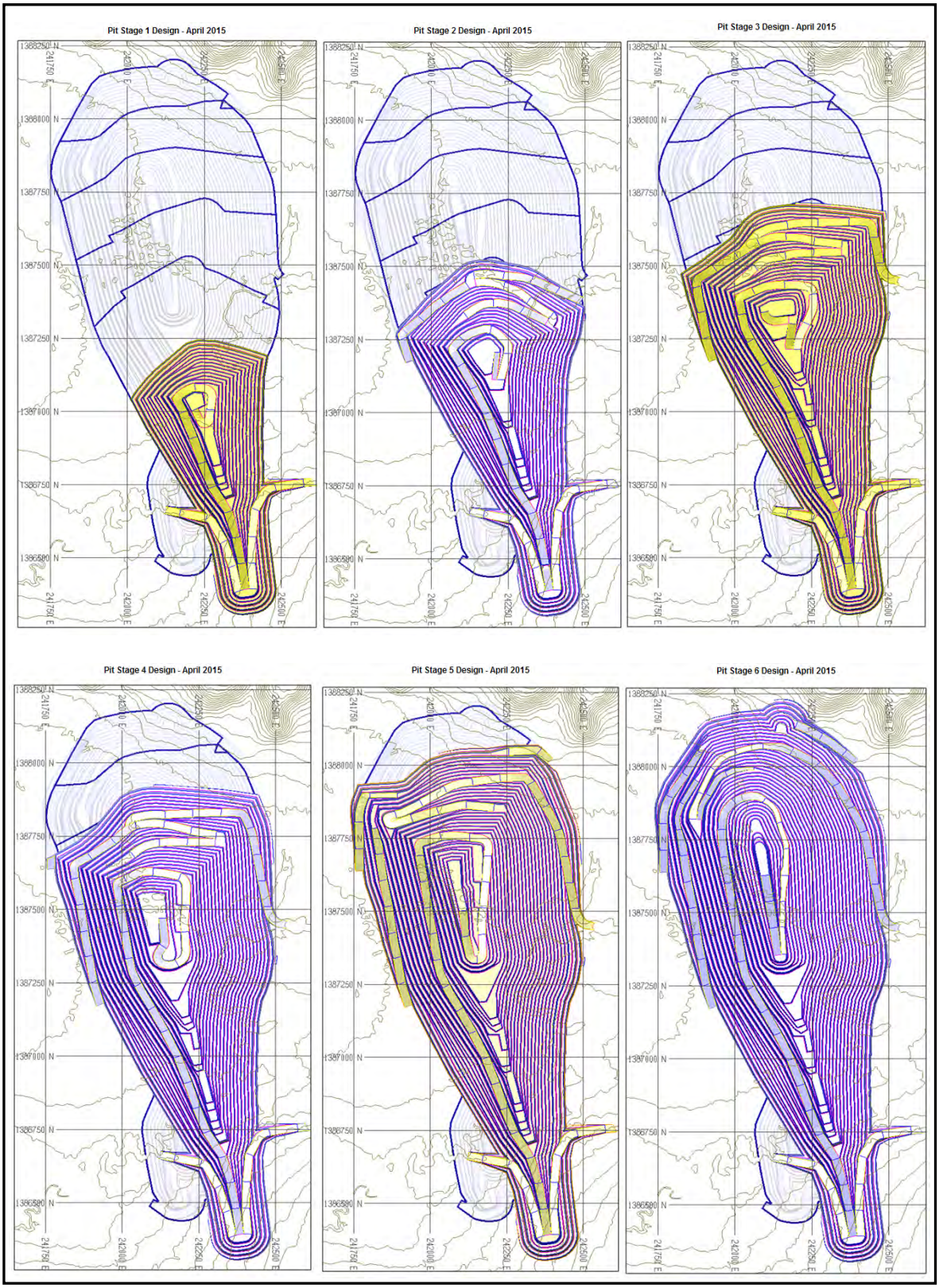


The Stage 2 cutback (Figure 15.6.4) will initially be accessed via two temporary ramps developed at the north wall again for ore haulage to the east and waste haulage to the west. The temporary ramps will cut-off at approximately at 100 m depth as the lower levels of Stage 2 pit will be accessible via the main ramp developed in Stage 1.

The above pattern for Stage 2 ramp system will be repeated in the development of further cutbacks to the north as seen in Figure 15.6.4.

Both the temporary and main ramp systems will be developed deeper to access the mining of the lower levels. The access to the lower levels of the earlier cutbacks will be maintained while the new cutbacks developed.

Figure 15.6.4 Northern Expansion of the Pit Limits in Six Stages



15.7 Pit Stage Mining Inventory

15.7.1 Cut-off Grades for Ore Classification

The 1.1 g/t mill feed cut-off grade was determined as optimal for the early production years based on NPV analyses in pit optimisations. The higher cut-off grade improved the operating NPV of the project significantly (~8%) compared to processing at the break-even cut-off grade.

The cut-off grades used to classify the ore types in the open pit mining inventory and production schedules are as follows:

- The break-even cut-off grade is 0.8 g/t gold as used to classify the Mineral Reserves.
- The ore between 0.8 g/t and 1.1 g/t gold is classified as low grade (LG) for stockpiling and processing at the end of the mine life.
- The ore between 1.1 - 1.5 g/t cut-off is classified as medium grade (MG) for processing as necessary to meet processing feed tonnage requirements, along with the high grade ROM ore.
- The ore above 1.5 g/t grade is classified as high grade (HG) for ROM processing over the mine life.

The mill feed cut-off grade will vary through mine life depending on the availability of the ore stocks and grades mined in the cutbacks.

15.7.2 Mining Inventory by Cut-off Grade and Pit Stages

Table 15.7.1 below summarises pit stage mining inventory for the ore types used in the production schedules. The estimated production cost per recovered ounce increases from US$ 450/oz in the first pit stage to US$ 810/oz in the Stage 6 cutback as the stripping ratio increases. The estimated production cost is highest at US$ 1,000/oz for the Stage 7 cutback, due to the significantly lower (1.44 g/t) grade of the mined ore.

The review of continuity of the high grade core and the resource grade-tonnage distribution showed that the cut-off grades up to 1.5 g/t can be sustained for selective mining on the mining benches. The exception to this will be the low grade mineralisation mined in Stage 7, which will not be an issue as the mill feed cut-off grade will be lower and the lower grade stockpiles will be blended towards the end of mine life.

Table 15.7.1 Mining Inventory Summary by Pit Stages

Pit Stage	Max. Depth	Total	SR	Waste[1]	Total Ore		LG Ore[2]		MG Ore[3]		HG Ore[4]		Gold (koz)[5]		Prod.[6,7] Cost
		kt	tt	kt	kt	Au g/t	kt	Au g/t	kt	Au g/t	kt	Au g/t	Cont'd	Recov'd	$/oz
Total	320	268,545	4.5	219,373	49,173	2.35	8,865	0.90	11,765	1.25	28,543	3.26	3,718	3,450	590
1	150	37,203	2.7	27,163	10,040	2.71	1,724	0.90	2,050	1.24	6,266	3.69	874	811	450
2	190	37,355	3.3	28,609	8,745	2.27	1,492	0.91	2,253	1.25	5,000	3.14	639	593	560
3	230	46,924	3.6	36,782	10,142	2.30	1,824	0.91	2,573	1.26	5,744	3.20	749	695	570
4	260	58,374	4.9	48,479	9,895	2.40	1,858	0.90	2,295	1.24	5,742	3.36	765	710	590
5	290	44,340	7.9	39,348	4,992	2.18	874	0.90	1,250	1.26	2,868	2.97	350	325	790
6	320	36,304	8.1	32,313	3,991	2.17	678	0.90	953	1.25	2,360	2.90	278	258	810
7	120	8,045	4.9	6,678	1,367	1.44	415	0.90	391	1.24	561	1.96	63	58	1,000

1. **Waste classification includes the sub-grade material reported between 0.5 to 0.8 g/t gold grade.**
2. **LG Ore – Low Grade Ore classification is between 0.8 to 1.1 g/t gold grade.**
3. **MG Ore – Medium Grade Ore classification is between 1.1 to 1.5 g/t gold grade.**
4. **HG Ore – High Grade Ore classification is above 1.5 g/t gold grade.**
5. **An average recovery of 92.8% was used to calculate the recovered ounces.**
6. **The reported production cost per ounce is based on average unit costs applied to pit design and optimization for comparison purposes. Note that this is similar to but not the same as the final operating and production costs (see Sections 21 and 22).**
7. **The operating cost is reported, including preproduction prestrip and excluding royalty and capital costs.**

15.8 Waste Storage Facility Designs and Site Layout

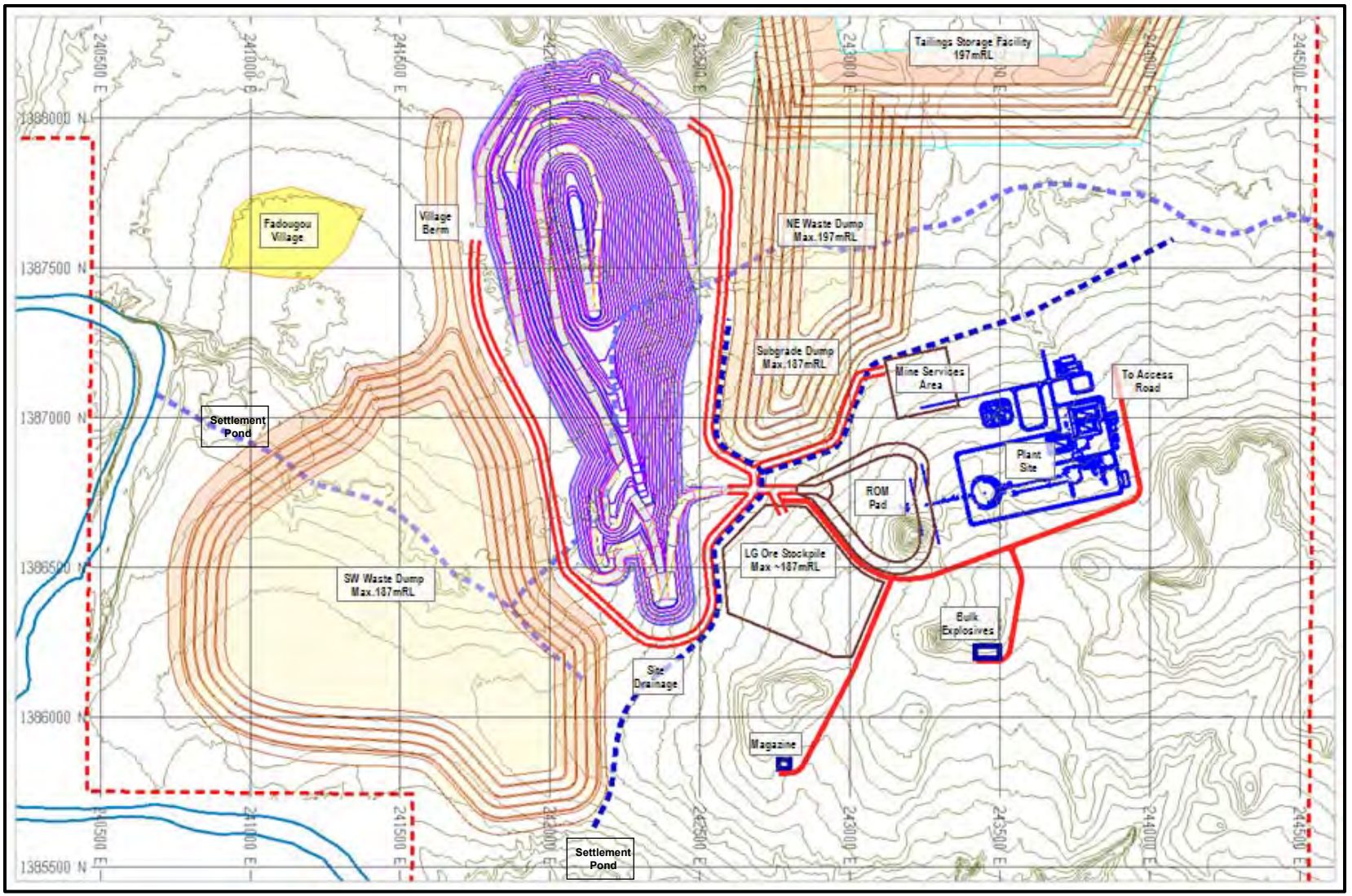
The proposed site plan in Figure 15.8.1 includes the extents of the ultimate pit design, waste storage facility designs, mine roads, stockpile areas, ROM pad, and mining service area. The site plan and infrastructure requirements will be detailed in the finalisation of the OFS.

The TSF embankment design, mining structures, roads, and drainage layout were integrated at the east of the proposed site plan for better utilisation of the available area, reducing the site establishment and maintenance requirements.

The water drainage from the waste storage facilities, stockpiles and spill from the open pits will be controlled through two settlement ponds before discharging to the river at the west. The settlement ponds will be located on the proposed drainage channel at the south and in the small catchment to the northwest of the southwest waste facility.

All the waste mined from the earlier cutbacks will be hauled to the southwest facility except the waste required for the construction of the early stages of the TSF embankment. The smaller east waste storage facility and the TSF embankment will be reserved mostly for the waste mined from the upper levels of the later pit stages at the north. All the waste mined from the lower levels of the cutbacks will be hauled to the southwest waste facility through the main ramp.

Figure 15.8.1 Waste Storage Facility Designs and Site Layout



15.9 Risk Assessment

The risks have been assessed separately for the development (technical) and operations stage of the Project considering the different nature of the events. The risks for the operations stage have been assessed at the end of Section 16.

Table 15.9.1 provides the risk assessment for the development stage of the Project. The issues / events considered are mainly related to the investigations, data quality, estimations, planning, and decision making.

Table 15.9.1 Project Development Risk Assessment

Failure Event	Likelihood of Occurrence	Potential Severity of Impact	Risk Rank
Over-estimation of mineral resources	Possible	Moderate	High
Under-estimate of mining conditions	Unlikely	Moderate	Medium
Under-estimate of mining costs for reserve estimate	Possible	Minor	Medium
Under-estimate of processing & administration costs	Possible	Minor	Medium
Over-estimate of metallurgical performance for reserves estimate	Unlikely	Moderate	Medium
Over-estimate of gold price for reserves estimate	Possible	Moderate	High
Pit slopes are too steep	Possible	Minor	Medium
Over-estimate of ore reserves	Possible	Moderate	High
Feasibility design is not fully achievable	Possible	Minor	Medium
Unsuitable mining method	Unlikely	Minor	Low
Production schedule is not achievable	Unlikely	Moderate	Medium

None of the high risks identified are considered as fatal flaws for the Project with “minor” to “moderate” impacts, due to the relatively favourable characteristics of the Fekola deposit. The pit optimisation sensitivity analysis results in Section 15.4.3 provide a quantitative assessment of the potential impacts of changing conditions on the project.

The pit limits are not sensitive to change in the study parameters except the last two pit stages containing 11% (5.5 Mt) of Mineral Reserves. These stages will be committed in the fifth year of operations, well after the capital payback period of approximately 2.25 years.

The strip ratio is relatively low for the first four pit stages, containing 80% (39 Mt) of the Mineral Reserve tonnage and 80% (3 Moz) of the Mineral Reserve ounces, and is generally low (4.5:1 t/t) overall. The comparative unit costs (US$/oz) reported for the pit stages show that the first four pit stages would be viable economically to mine for gold prices down to US$ 700/oz.

To mitigate the potential impacts, the high risks identified for the OFS will require monitoring and preparation in the operations stage, especially during the early years of the operations.

It can be concluded that there are no impacts currently identified that would prevent the project going ahead. Provided that the monitoring of the conditions and accordingly planning of the operations are carried out, there are no impacts currently identified that would affect the mining plans and the Mineral Reserve estimates materially.

FEKOLA GOLD PROJECT

NI 43-101 - MINING METHODS

3174-STY-002

Table of Contents

			Page
16.0	**MINING**		**16.1**
	16.1	Mining Method	16.1
		16.1.1 General Definition	16.1
		16.1.2 Shifts and Personnel	16.1
		16.1.3 Mining Equipment	16.2
		16.1.4 Grade Control	16.3
		16.1.5 Ore Handling	16.3
	16.2	Drilling Blasting	16.4
	16.3	Pit Dewatering	16.4
	16.4	Environmental Monitoring and Control	16.5
	16.5	Mine Production Schedule	16.5
	16.6	Mining Risk Assessment	16.9
	16.7	Limitations and Recommendations	16.9

TABLES

Table 16.1.1	Mine Personnel Requirements	16.2
Table 16.1.2	Mine Equipment Requirements	16.3
Table 16.2.1	Powder Factor Estimates (ANFO Equivalent)	16.4
Table 16.5.1	Production Schedule Overview	16.6
Table 16.5.2	Base Case Production Schedule Annual Summary	16.7
Table 16.6.1	Mining Operations Risk Assessment	16.9

FIGURES

Figure 16.5.1	Production Schedule Summary Graphs	16.8

16.0 MINING

16.1 Mining Method

16.1.1 General Definition

The selected mining method for the Project is conventional open pit mining with all operations carried out by B2Gold. All values and results reported are on a 100% B2Gold ownership basis.

The open pit mine life is 9.5 years for the development of 320 m deep ultimate pit in a sequence of seven stages (cutbacks). The base case mine production schedule involves the movement of 32 Mtpa material to sustain processing of 4 Mtpa of high grade (~3 g/t gold) ore. The processing of the 14 Mt of stockpiled low grade (~1 g/t gold) ore is scheduled for processing towards the end of the mine life.

As upside potential B2Gold considers that part of the stockpiled low and medium grade ore can potentially be processed in the ROM feed at marginal cost, as the processing plant was designed for the 4 Mtpa nameplate capacity with a 25% (1 Mtpa) design factor.

In general, three cutbacks will be mined simultaneously to expose sufficient ore stocks to sustain the plant feed through mine life. Mostly ore will be mined from the most advanced cutback at depth. The second cutback will be catching up to expose more ore at depth as the waste pre-stripping is completed. Mostly waste will be mined in the third cutback at the upper levels to expose future ore on time. This pattern will repeat as the mining in the first cutback is completed and replaced with the ore mining in the second cutback and a new cutback is started at the top of the open pit.

16.1.2 Shifts and Personnel

The mining operations are scheduled to work 330 days in a year. It is assumed that 35 days production in a year will be lost due to unscheduled delays, such as those occurring during the high rainfall events. The processing plant is scheduled to operate 24 hours continuously except for planned maintenance periods. The mill feed will be supplied from the ROM ore stockpiles while the mine is not producing ore feed.

The mining workforce will be accommodated in the surrounding villages and towns, and be bussed to and from the mine on a shift basis. The senior staff will be accommodated in the village established as part of the mine site.

It is assumed in the manpower estimates in Table 16.1.1 that the mine operations will be based on 3 x 8 hour shifts. Sufficient standby personnel were allowed for the roster off and other leaves. The number of expatriate staff will be reduced over time as the local staff is developed.

The majority of the mining administration, supervision and technical personnel listed in Table 16.1.1 will work on the day shift on a 2 on 1 off basis.

Table 16.1.1 Mine Personnel Requirements

Labour Category	Day Shift	Total
Total Labour	**211**	**573**
Skilled and Unskilled Labour	**176**	**506**
Load, Haul, and Ancillary	59	234
Drilling and Blasting	35	90
Maintenance	60	140
Health and Safety	2	6
Technical Services	12	18
Other Labour	8	18
Senior Staff	**20**	**42**
Load, Haul, and Ancillary	2	8
Drilling and Blasting	3	7
Maintenance	7	13
Health and Safety	1	2
Production Administration	4	8
Technical Services / Administration	3	4
Expatriate Staff	**15**	**25**
General Administration	2	3
Mine Production	2	3
Drilling and Blasting	1	2
Maintenance	5	10
Health and Safety	1	1
Purchasing	2	3
Technical Services / Administration	2	3

16.1.3 Mining Equipment

Table 16.1.2 lists the start-up and peak mining equipment requirements for the Project during the first eight years of production. The truck numbers will increase from 16 in the pre-stripping period to 34 as the haul distances increase with the deepening of the mined cutbacks. The excavator numbers will peak in the ramp up period early in the production schedule with addition of six trucks to the initial fleet at the start of mining in Stage 2.

The three PC2000 (or similar) shovels will work on 10m high waste mining benches. The two PC1250 (or similar) backhoes will be mainly used to mine ore selectively and to mine difficult areas. The ancillary equipment numbers will peak at the end of second year of the schedule as the mining in the three cutbacks and waste dumping faces advance.

The proposed equipment fleet will provide relative flexibility in the utilisation as three pit stages will be mined simultaneously to mine waste and ore at different levels.

Table 16.1.2 Mine Equipment Requirements

Mining Equipment	Start-up Numbers	Peak Numbers
Excavator PC2000 – Shovel	2	3
Excavator PC1250 – Backhoe	2	2
Trucks (777D/G)	16	34
Cat D9R Dozer	3	4
Cat 844K Wheel Dozer	1	1
Cat 16M Grader	2	3
Wheel Loader 988K	1	2
Wheel Loader 966K	1	2
Cat 773D WT Watertruck	2	3
Cat 330E Excav/break	1	1
Large DHH Drills	4	5
TH Drills	2	3
RC Drill	1	2

16.1.4 Grade Control

In grade control of the mining benches, the sampling will commence with the RC drilling ahead of the mining front to assist the short and medium term mine planning processes. The holes will be angled at 60 degrees from the hanging wall side of the ore zones to provide a good intersection with the mineralised structures.

The grade control will be based on drilling of 140 mm diameter 20 m long RC holes and sampling practice at 1.5 m intervals along the holes. The samples will be tested in the onsite laboratory. A three stage grade control programme was proposed for the project as follows:

- 20 m x 40 m pattern infill drilling of the entire mining benches.
- 10 m x 20 m pattern drilling in the zones identified for ore recovery.
- 5 m x 10 m pattern drilling in the zones with less continuity (usually peripheral zones with higher grade variability).

The proposed grade control programme provides data coverage of approximately three months in advance with some overlap between the drilled levels.

16.1.5 Ore Handling

The mill feed ore will be transported from open pits to the ROM pad for direct tipping or stockpiling. It is assumed that 50% of the ROM feed needs to be stockpiled to regulate the mine production and crusher feed rates.

The stockpiled ore will be loaded to the crusher with a Cat 988K loader or equivalent. It is planned to maintain approximately three to four week's ore supply on the ROM pad in the form of stockpile fingers to control the head grades.

16.2 Drilling Blasting

The production drilling, blasting, and waste mining operations will be carried out in general at 10 m benches. It is expected, as required by geological conditions, that 5 m bench blasting will be required in places for mining at two 2.5 m flitches to achieve a higher degree of selectivity.

The powder factor estimates in Table 16.2.1 depends on the weathering degree, rock strength and associated rock densities. The weighted average powder factor is calculated to be 0.75 kg/bcm (ANFO Equivalent) for the total volume mined.

Table 16.2.1 Powder Factor Estimates (ANFO Equivalent)

Weathering Degree	Bulk Density (t/bcm)	Suggested Powder Factor (kg/bcm)
Weathered (with lateritised layers)	1.80 (2.40)	0.40 (Average)
Transition (weather and fresh mix)	2.40	0.60 (Average)
Fresh Rock	2.77	0.80 (0.65 – 1.00)

Five large and three small drills will be required for the drill and blast operations, based on the powder factor and drilling productivity estimates. It is assumed that bulk emulsion will be used throughout the operations under both dry and mostly wet conditions.

For blasting at 10 m benches, the 165 mm diameter blast hole patterns are expected to be varying from 4 m x 5 m in fresh rock to 4.8 m x 6 m in transitional rock, depending on the variability and strength of the formations.

For blasting at 5 m benches, the 115 m diameter blast hole patterns are expected to be varying from 2.7 m x 3.4 m in fresh rock to 3.2 m x 4 m in transition rock, depending on the variability and strength of the formations.

Free digging, ripping, and blasting operations in the weathered zone will vary according to the extent of the lateritised zones and potentially presence of boulders. A small excavator with rock breaker attachment may be required for the fragmentation of the oversize in the pit.

16.3 Pit Dewatering

Results of hydrogeological investigations indicate that the pre-mining groundwater table is located at depths of between 2 m and 5 m around the proposed pit perimeter. It is currently assumed that mining operations will take place under wet conditions with borehole and in-pit de-watering programmes in place.

It is expected that the water inflows will be higher in the first 20 to 30 m depth of the open pit development and mining through the broken rock zones around the pit perimeter.

Based on the results and limitations of the hydrogeological investigations, a groundwater inflow rate of 170 l/s is predicted for the Fekola open pit at the end of Year 1, gradually increasing to 250 l/s at the end of Year 4. The inflow rates can be up to 410 l/s at the end of Year 4 in the worst case scenario.

Allowances have been made in the capital and operating costs for the installation of the dewatering bore holes and in-pit dewatering. In addition to the two existing 150 m and 160 m deep holes, nine boreholes were planned in the de-watering bore schedule with the depths varying from 180 m to 360 m.

Groundwater level monitoring in the regional bores will continue and will also be undertaken in the dewatering bores. Drawdown in the early phase of dewatering will be carefully monitored and managed via the recording of abstraction rates.

16.4 Environmental Monitoring and Control

Monitoring the environmental impacts and mitigation, especially the effects on the local population living in the Fadougou village, will be the key objective of the Environmental and Social Management Plan and its implementation.

The monitoring of the air and water quality and generation of noise and vibrations will be conducted on a regular basis. The monitoring data will be evaluated and appropriate mitigation measures will be implemented as needed.

Environmentally hazardous artificial materials, such as chemicals and rubbish, will be disposed following the appropriate standard procedures. Dust from the pits, dumps, stockpile operations, and road surfaces will be controlled by water suppression.

The surface water from the pit and waste dumps will be treated in the settlement ponds as necessary, before releasing to the environment (or using in the plant). The waste facilities and long-term stockpiles will be built to improve the stability and surface water drainage to minimise the environmental impacts.

The final faces of the waste facilities will be dozed to the specified slope angles and contoured for rehabilitation. The rehabilitation of the waste facilities will be carried out progressively as the final faces established around the perimeter.

16.5 Mine Production Schedule

The mine production schedules for the OFS were prepared on a monthly basis for the first two years of the operation and quarterly for the rest of the mine life.

An overall summary of the base case production schedule is presented in Table 16.5.1, followed by the annual summary in Table 16.5.2.

The major constraints applied in the production schedule can be summarised as follows:

- Maintaining relatively constant working strip ratios and smoother mining rates for better utilisation of the mining equipment though mine life.
- 15% lower mine production rate during the rainy season (five months from mid June to mid November).
- Maintaining approximately four week's of mill feed stockpile on the ROM pad at reasonable grade depending on the availability of ore in the pit.
- Keeping the average vertical mining advance (sink) rates generally below 60 m/year.

The sink rates will be relatively low below 40 m/year during the mining of the earlier pit stages in the first five years of the operations. The sink rates later in mine life will increase up to 75 m/year in mining waste during the development of the deeper pit Stages 5 and 6.

The graphs in Figure 16.5.1 show that the total tonnes mined annually are constant at 32 Mtpa in the first seven years of the production schedule. The mine production tails off in the last 2.5 years as the pre-stripping of the last pit stages completed. The scheduled feed grade is on average 0.7 g/t higher than the mined grade in the first seven years, due to the low grade stockpiling strategy. The scheduled ore types show that the mill feed cut-off grade of 1.5 g/t in the first two years gradually drops to 1.1 g/t in the later years, and 0.8 g/t at the end of the mine life.

Evaluation of the production schedule options showed that the ramping up the throughput rate from 4 Mtpa nameplate capacity, to 5 Mtpa, improves the operating NPV of the Project by approximately US$ 100 M (~6.5%).

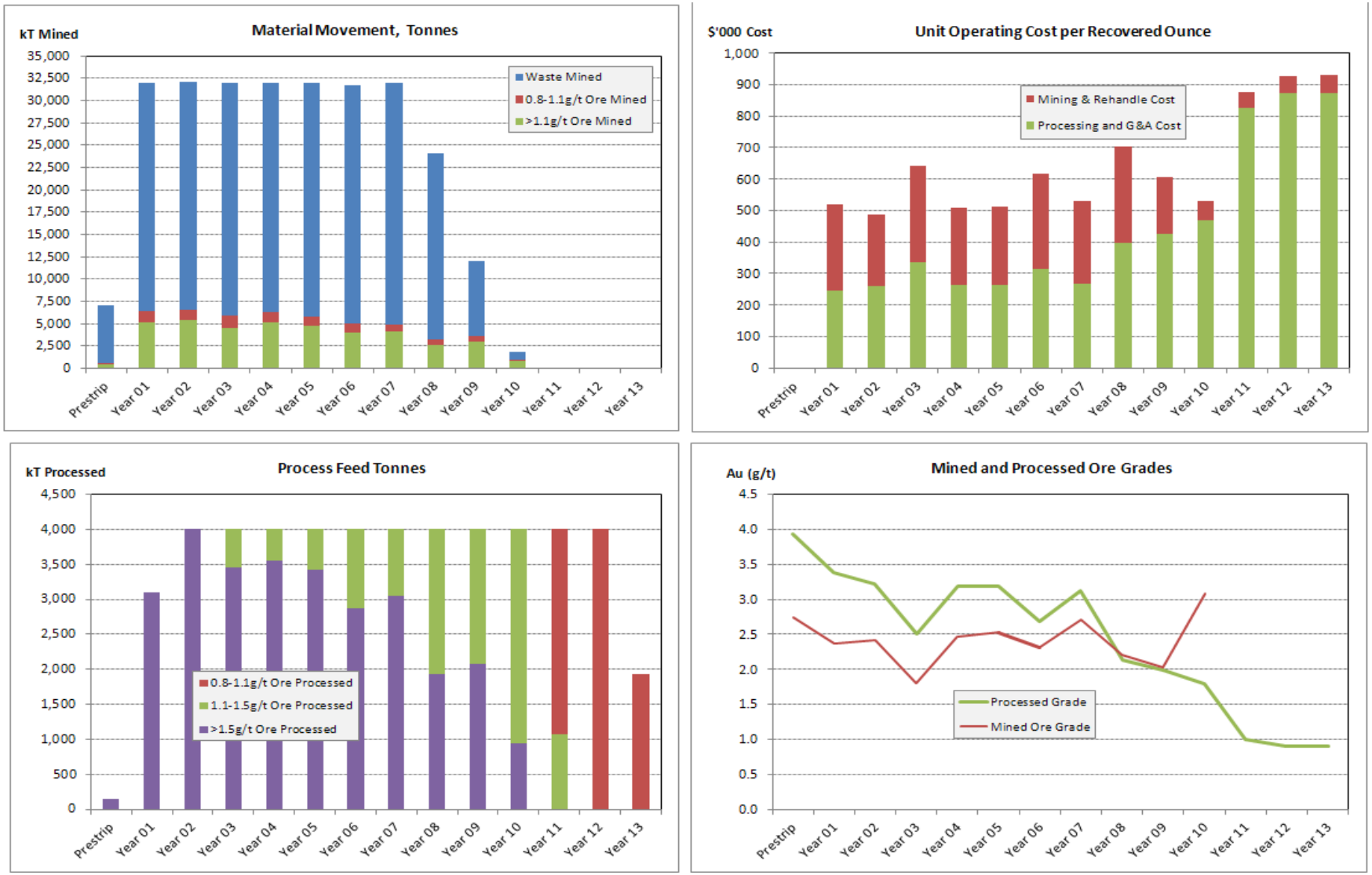
Table 16.5.1 Production Schedule Overview

Pre-stripping period (with production ramp-up)	~3 months
Open pit mine life (including pre-strip)	9.5 years
Open pit nominal production rate	32 Mtpa
Processing plant life	12.5 years
Processing rate	4 Mtpa
Average mined ore grade (first 7 Years)	2.4 g/t Au
Average mill feed ore grade (first 7 Years)	3.1 g/t Au
Maximum long-term stockpiled ore tonnage	14 Mt
Long-term stockpiled ore grade (7 years)	1.1 g/t Au
Average ROM gold production (first 7 years)	350 koz pa
Average ROM and stockpile gold production (8 – 10 years)	235 koz pa
Low grade stockpile gold production (last 2 years)	90 koz pa

Table 16.5.2 Base Case Production Schedule Annual Summary

Production Schedule 1b v3 Summary 4Mtpa Processing, 32Mtpa Mining		Total	2017 Prestrip	2018 Year 01	2019 Year 02	2020 Year 03	2021 Year 04	2022 Year 05	2023 Year 06	2024 Year 07	2025 Year 08	2026 Year 09	2027 Year 10	2028 Year 11	2029 Year 12	2030 Year 13
Physicals																
Waste Mined	kBCM	84,264	3,341	10,306	9,853	10,309	9,405	10,038	9,894	10,039	7,686	3,052	340			
Ore Mined	kBCM	17,916	283	2,365	2,373	2,131	2,259	2,092	1,801	1,784	1,185	1,298	343			
Total Mined	kBCM	102,180	3,625	12,672	12,226	12,440	11,664	12,130	11,696	11,823	8,871	4,351	683			
Waste Mined	kT	219,373	6,427	25,560	25,523	26,085	25,718	26,127	26,772	27,000	20,763	8,455	943			
Ore Mined	kT	49,173	602	6,400	6,531	5,858	6,258	5,791	4,980	4,927	3,278	3,597	949			
Au Grade	g/t	2.35	2.74	2.37	2.42	1.81	2.47	2.53	2.31	2.72	2.21	2.03	3.08			
Total Mined	kT	268,545	7,029	31,960	32,054	31,943	31,975	31,919	31,753	31,927	24,041	12,052	1,892			
	kbcm	102,180	3,625	12,672	12,226	12,440	11,664	12,130	11,696	11,823	8,871	4,351	683			
Ore Stockpiled	kT	16,439	458	3,374	2,737	2,367	2,258	1,857	1,029	1,048	631	609	71			
Stockpile Closing Balance	kT	14,105	458	3,760	6,291	8,148	10,406	12,198	13,178	14,105	13,384	12,980	9,930	5,930	1,930	0
Ore Reclaimed	kT	16,439		73	206	510		65	48	121	1,352	1,012	3,122	4,000	4,000	1,930
Total Process Feed	kT	49,173	144	3,099	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	1,930
	Au g/t	2.35	3.94	3.38	3.23	2.50	3.19	3.18	2.68	3.12	2.13	1.99	1.80	0.99	0.90	0.90
Gold Ounces																
Mill Feed	koz	3,718	18	337	415	322	410	410	345	401	274	256	231	128	116	56
Recovered	koz	3,450	17	314	387	299	382	381	320	374	253	236	213	117	106	51
Recovery	%	92.8%	93.5%	93.3%	93.2%	92.8%	93.2%	93.2%	92.9%	93.1%	92.5%	92.4%	92.3%	91.2%	91.0%	91.0%

Figure 16.5.1 Production Schedule Summary Graphs



16.6 Mining Risk Assessment

The risk assessment for the operations stage of the Project has been provided in Table 16.6.1. The monitoring and analysis of the production environment and the historical data will be critical for the continuous improvement of the operations as the mining progresses. The safety of the personnel, achieving the production targets and management of social and environmental impacts take precedence in the operations stage.

Table 16.6.1 Mining Operations Risk Assessment

Failure Event	Likelihood of Occurrence	Potential Severity of Impact	Risk Rank
Mining equipment productivities are slow, slow ramp-up at the start	Unlikely	Moderate	Medium
Final pit design is not achievable	Possible	Moderate	High
Grade control practice or resource model under-performs	Possible	Minor	Medium
Climate stopping mining (other than planned)	Possible	Minor	Medium
Water drainage management, run-off control issues	Possible	Moderate	High
Under-estimation of pit ground water inflows	Possible	Moderate	High
Noise management issues (after mitigation)	Possible	Minor	Medium
Minor pit wall failure	Likely	Minor	Medium
Major pit wall failure	Unlikely	Major	High
Waste dump stability issues	Unlikely	Minor	Low
Local public relation issues (after mitigation)	Possible	Minor	Medium

None of the high risks identified are considered as fatal flaws for the Project with “minor” to “moderate” impacts, since the Fekola pit will be developed progressively deeper along strike in six stages (seven stages with shallower cutback at southwest). The cutback areas are relatively large for the operation of the equipment to sustain the scheduled production rates. The vertical advance (sink) rates are below 40 m per year in the early years of mining.

There will be opportunity in the early years of the open pit life to monitor, evaluate, and adjust the mining plans and operations before committing on the final pit limits. The pit optimisation sensitivity analysis results in Section 15.4.2 and the mining study limitations and recommendations in Section 16.7 below provide further information for the impact and mitigation of the identified risks for the project.

Provided that the required monitoring of the conditions and planning of the operations are carried out especially in the early years of the mine life, there are no impacts currently identified that would affect the mining plans and open pit operations materially.

16.7 Limitations and Recommendations

The OFS is based on long term plans to highlight any critical operational issues and provide cost estimates for the life of mine cash flow models. Detailed planning of the roads and other site layout will be required before the implementation stage.

Monitoring the environmental impacts and mitigation, especially the effects on the local population living in the Fadougou village, will be the key objective of the Environmental and Social Management Plan and its implementation. The drilling and blasting operations at the upper levels of the later pit stages closer to the Fadougou village will require advanced planning and closer control based on the experience gained in the earlier pit stages.

It will be critical for planning the in-pit dewatering that the borehole and in-pit dewatering abstraction rates are monitored especially in the early stages of dewatering. Groundwater level monitoring in the regional bores should continue and should also be undertaken in the dewatering bores.

Additional geotechnical work will be required prior to and during mining. Geological wall mapping (including structural geological measurements), and ongoing assessments of wall stability will be required over the operational life of the pits. Periodic reviews of wall stability will need to be conducted. Results of this work will allow any future amendments to local wall designs required to maintain stability to be put into effect in a timely manner.

Grade control will be required in advance and regular reconciliation of the estimated tonnages and grades in the mining plans to actual mill production will be required for the adjustment of the short and medium term mining plans.

Short to medium term planning of the mining operations will ensure that there is sufficient in-pit and external stockpiles available before the rainy seasons. The utilisation of the mining equipment in the cutbacks can also be scheduled at higher elevations during the rainy seasons.

Considering the relatively good continuity of the ore zones at Fekola, a standard ~250 t excavator (backhoe) fleet can be a viable alternative to currently selected excavator fleet. This option can improve the productivity per loading and hauling capacity, reduce the required equipment numbers slightly, and facilitate the scheduling of the standard equipment in the multiple cutbacks. However, mining at maximum 5 m benches and potentially better rock fragmentation requirements (beneficial to lower the overall cost in any case) for this option will require closer control and planning of the open pit operations.

FEKOLA GOLD PROJECT

NI 43-101 - RECOVERY METHODS

3174-STY-002

Table of Contents

			Page
17.0	**RECOVERY METHODS**		**17.1**
	17.1	Process Design	17.1
		17.1.1 Selected Process Flowsheet	17.1
		17.1.2 Key Process Design Criteria	17.4
	17.2	Process and Plant Description	17.5
		17.2.1 Ore Receiving and Crushing	17.5
		17.2.2 Crushed Ore Stockpile	17.6
		17.2.3 Grinding and Classification	17.6
		17.2.4 Pebble Crushing	17.8
		17.2.5 Leach Thickening	17.8
		17.2.6 Carbon in Columns Circuit	17.9
		17.2.7 Leach Circuit	17.10
		17.2.8 Carbon in Pulp Circuit	17.11
		17.2.9 Acid Wash, Elution, Electrowinning and Gold Room	17.12
		17.2.10 Carbon Regeneration	17.15
		17.2.11 Cyanide Destruction	17.15
		17.2.12 Tailings Thickening and Disposal	17.16
		17.2.13 Reagents Mixing and Storage	17.16
		17.2.14 Water Services	17.20
		17.2.15 Air Services	17.22
	17.3	Plant Consumption	17.22
		17.3.1 Energy Consumption	17.22
		17.3.2 Water Consumption	17.23
		17.3.3 Reagent and Consumable Consumption	17.23
	17.4	Plant Control System	17.24

TABLES

Table 17.1.1	Summary of Key Process Design Criteria	17.4
Table 17.3.1	Average Power Demand Summary	17.23
Table 17.3.2	Annual Reagent and Consumable Consumption	17.24

FIGURES

Figure 17.1.1	Overall Process Flow Diagram	17.3

17.0 RECOVERY METHODS

17.1 Process Design

The process plant design for the Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is based upon unit operations that are well proven in industry.

The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project and ore specific criteria that the plant design must meet are:

- 4 Mtpa of ore.
- All major unit operations and equipment will be sized with a 25% design margin so the plant is expandable to a maximum throughput of 5 Mtpa without any further design margin.
- Process plant availability of 94% supported by crushed ore storage, standby equipment in critical areas and on-site HFO and diesel generator power supply.
- Sufficient automated plant control to minimise the need for continuous operator interface and allow manual override and control if and when required.

Study design documents have been prepared incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical testwork.

17.1.1 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Single stage primary crushing with a gyratory crusher to produce a crushed product size of 80% passing (P80) of 150 mm.
- Crushed ore stockpile with a nominal 10,000 tonne live capacity to provide 20 hours of operation at design plant throughput. During extended periods of up to three days for primary crusher equipment maintenance, ore from the dead part of the stockpile is reclaimed by an excavator or dozer to feed the grinding circuit.
- Crushed ore from the stockpile is normally reclaimed by apron feeders positioned under the stockpile to feed the grinding circuit.

- The grinding circuit is a SABC type, which consists of an open circuit SAG mill, pebble crusher for SAG mill discharge oversize and a closed circuit ball mill to produce a P80 grind size of 75 µm.

- A gravity gold recovery circuit is not included as the metallurgical testwork results do not warrant it.

- Quicklime from a silo is added onto the SAG mill feed conveyor along with the crushed pebbles. Sodium cyanide solution is added to the SAG mill feed chute to start the gold leaching process.

- Hydrocyclones are operated to achieve a cyclone overflow slurry density of 25% solids to promote better particle size separation efficiency. Subsequently, a leach thickener is included to increase slurry density to the leach circuit, minimise leach tank volume requirements, reduce overall reagent consumption, and separate gold dissolved by cyanide addition to the grinding circuit.

- Carbon columns (CIC) are included to recover gold already dissolved in the grinding circuit. The leach thickener overflow stream is pumped to this carbon adsorption circuit.

- Leach circuit with five tanks to achieve the required 24 hours of residence time at design plant throughput. Carbon-in-pulp (CIP) circuit consisting of six stages is a carbon adsorption circuit for recovery of remaining gold dissolved in the leaching circuit.

- Zadra elution circuit with gold recovery to doré. The circuit includes an acid wash column to remove inorganic foulants from the carbon with hydrochloric acid. The single elution circuit is common for both carbon adsorption circuits.

- Carbon regeneration kiln to remove organic foulants from the carbon with heat. This piece of equipment is common for both carbon adsorption circuits.

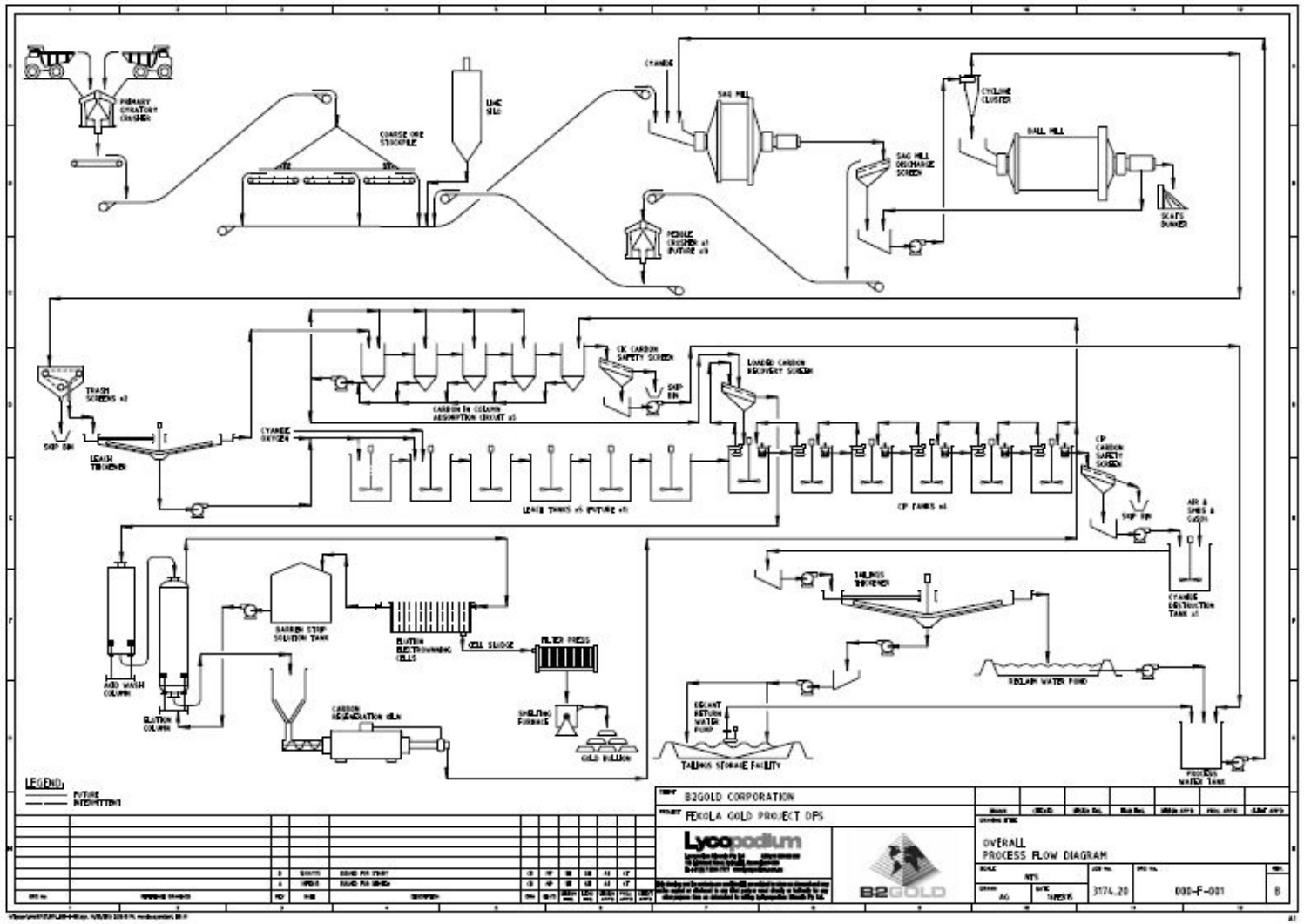
- Cyanide destruction circuit using SO_2 and air to reduce the WAD Cyanide level in the tailings discharge stream to an environmentally acceptable level.

- Tailings thickener to increase slurry density for water recovery prior to tailings discharge to the tailings storage facility.

An overall process flow diagram depicting the unit operations incorporated in the selected process flowsheet is presented in Figure 17.1.1.

The key issues considered in process and equipment selection are outlined in the following section.

Figure 17.1.1 Overall Process Flow Diagram



17.1.2 Key Process Design Criteria

The key process design criteria listed in Table 17.1.1 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17.1.1 Summary of Key Process Design Criteria

	Units	Design	Source*
Plant Throughput	tpa	4,000,000	Client
Head Grade	g Au/t	2.50	Client
Overall Gold Recovery[1]	%	92.75	Calc from Testwork
Crushing Plant Availability	%	65.0	Client
Plant Availability	%	94.0	Client
Crushing Work Index (CWi)	kWh/t	15.8	Testwork
Bond Rod Mill Work Index (RWi)	kWh/t	21.0	Testwork
Bond Ball Mill Work Index (BWi)[2]	kWh/t	20.3	Testwork
SMC Axb[3]		28.1	Consultant
Bond Abrasion Index (Ai)		0.703	Testwork
Grind Size	µm	75	Client
Leach Thickener Solids Loading	t/m2.h	1.39	Testwork
CIC Superficial Upflow Velocity	m/h	75	Lycopodium
Number of Carbon Columns (Stages)		5	Client
Leach Circuit Residence Time	hrs	24	Testwork
Leach Slurry Density	% solids (w/w)	45	Client
Number of Leach Tanks		5	Lycopodium
Number of Adsorption Tanks (Stages)		6	Lycopodium
Sodium Cyanide Addition	kg/t	0.78	Testwork
Lead Nitrate Addition	kg/t	0.10	Testwork
Dissolved Oxygen Level in Leach	ppm	13 - 17	Testwork
Quicklime Addition[4]	kg/t	0.74	Testwork
Elution Circuit Type		Zadra	Agreed
Elution Circuit Size	t	12	Agreed
Frequency of Elution	strips / week	7	Lycopodium
Cyanide Destruction Circuit Type		SO2 & Air	Testwork
SO2/ CNwad Weight Ratio	g SO2:g CNwad	4.0	Consultant
Tailings Thickener Solids Loading	t/m2.h	1.39	Testwork
Tailings Discharge Slurry Density	% solids (w/w)	60	Lycopodium

1. **At design head grade of 2.50 g Au/t.**
2. **Bond Ball Mill Work Index for design includes a 10% correction factor to the 85th percentile value for different results of comparable samples at different laboratories.**
3. **Design A x b value derived from the 85th percentile ranking of specific energies determined for each individual ore type.**
4. **Quicklime addition based on 90% CaO.**

17.2 Process and Plant Description

The overall process flowsheet includes a single stage gyratory crusher and a SABC grinding circuit with a ball mill in closed circuit with cyclones to achieve the final product size. The cyclone overflow stream will flow by gravity to two linear trash screens operating in parallel ahead of a leach thickener. Sodium cyanide is added to the SAG mill feed to start the gold leaching process. The leach thickener overflow solution is pumped to carbon columns to recover gold already dissolved in the grinding circuit. The thickened slurry is pumped to a leach circuit and then additional sodium cyanide along with lead nitrate and oxygen are added for further gold leaching. A carbon-in-pulp (CIP) circuit will adsorb dissolved gold onto activated carbon. A Zadra elution circuit will be used to recover gold from loaded carbon to produce doré. A cyanide destruction circuit using SO_2 and air will reduce the WAD cyanide level in the tailings stream to an environmentally acceptable level. The tailings stream is thickened to recover water before being pumped to the tailings storage facility.

17.2.1 Ore Receiving and Crushing

Run-of-mine (ROM) ore from the open pit, at maximum lump size of 900 mm, will be transported to the plant by 100 t capacity rear dump trucks. The trucks will tip directly into either side of the ROM pocket. However, if the trucks are not permitted to directly tip into the ROM pocket, then the truck load will be dumped onto the ROM pad. The ROM pad will be primarily utilised for emergency storage and ore blending if required. ROM ore will be reclaimed to the ROM pocket by a front-end loader.

A rock breaker will be installed to assist in breaking down, oversize material retained above the gyratory crusher in the ROM pocket. Ore will be crushed by the gyratory crusher and then withdrawn from the ROM discharge pocket by a variable speed apron feeder. A gyratory crusher has been selected on the basis that trucks can tip directly into it.

The crushed ore will be conveyed, via the stockpile feed conveyor, to the crushed ore stockpile. The stockpile feed conveyor will be fitted with a weightometer, to monitor the primary crusher throughput, and for controlling the apron feeder variable speed drive.

The crushing circuit will be serviced by a single dust collection system, consisting of multiple extraction hoods, ducting and a baghouse. Dust collected from this system will be discharged onto the stockpile feed conveyor and then finely sprayed with water for dust suppression.

Auxiliary equipment for the crushing circuit will include:

- Crushing area control room.
- Primary crusher grease pump.
- Primary crusher lubrication and hydroset pump systems.
- Primary crusher area cameras.

17.2.2 Crushed Ore Stockpile

The crushed ore stockpile will have a live capacity of approximately 10,000 t (equivalent to 20 hrs of mill feed at 4 Mtpa) and a total storage capacity of 71 hours.

Crushed ore will be reclaimed from the stockpile, by two of three variable speed apron feeders. The feeders will discharge onto the SAG mill feed conveyor which will convey the crushed ore to the SAG mill feed chute. The SAG mill feed conveyor will be fitted with a weightometer, used for controlling the speed of the reclaim feeders and for mass accounting of feed presented to the grinding circuit.

Quicklime will be added to the SAG mill feed conveyor in order to achieve the desired pH of 10.5 in the leaching circuit. This pH modification will maintain the protective alkalinity of the grinding and leaching circuits and minimise HCN gas evolution.

A ventilation fan will force air into the concrete stockpile reclaim chamber to ensure fresh air ventilates the upper part of the chamber which would otherwise have limited natural ventilation.

Auxiliary equipment for the reclaim area will include:

- Reclaim feeder lubrication unit.
- Stockpile and reclaim area cameras.

17.2.3 Grinding and Classification

The Fekola grinding circuit will be a traditional SABC circuit, comprised of a single, variable speed, semi-autogenous grinding (SAG) mill and a single fixed speed ball mill. The SAG mill will operate in closed circuit with a pebble crusher, whilst the ball mill will operate in closed circuit with hydro-cyclones. The product particle size exiting the grinding circuit (cyclone overflow) will contain 80% passing 75 µm material.

To meet the design throughput, and achieve the required leach product size, a 10.97 m x 5.33 m SAG mill (36 ft x 17.5 ft; 12 MW) and a 7.3 m x 10.8 m (24ft x 35.4ft; 10.3 MW) ball mill will be required.

Crushed ore, reclaimed from the stockpile, will be conveyed to the SAG mill feed chute via the SAG mill feed conveyor. Quicklime will also be added to the SAG mill feed conveyor before being conveyed to the SAG mill feed chute. Process water will be added to the SAG mill feed chute, to control the in-mill pulp density. Sodium cyanide will also be added to the SAG mill feed chute, to start gold leaching of the ore. The SAG mill will be fitted with discharge grates which will allow slurry to pass through the mill and will also relieve the mill of pebble build-up. The SAG mill product will discharge to a single deck vibrating screen, for pebble dewatering.

Grinding media (125 mm balls) will be added to the SAG mill via the SAG mill feed chute, utilising a dedicated media hoist, kibble and feed chute.

Sodium cyanide solution will be added to the SAG mill via the SAG mill feed chute, utilising a dedicated metering pump. The addition of sodium cyanide will start the gold leaching process.

SAG mill discharge screen oversize will be conveyed to a pebble crushing circuit. Undersize from the discharge screen will flow by gravity to the cyclone feed pump box, where it will combine with the discharge slurry from the ball mill. The slurry will then be pumped to the cyclone cluster by one of two (duty/standby) variable-speed cyclone feed pumps. Process water will be added to the cyclone feed pump box for cyclone feed density control.

The cyclone cluster overflow will flow by gravity through a metallurgical sampler then onto two linear trash screens in a parallel configuration. Two linear trash screens have been selected to provide sufficient area for a square aperture screen. The trash screen undersize will be directed to the leach thickener feed whilst trash screen oversize will be discharged to trash dewatering screens so the trash can be collected in a trash bin and the water can flow to the nearest mill area sump pump. Slurry from the cyclone underflow launder, will be returned to the ball mill feed chute with optional underflow slurry recycle to the SAG mill. Ball mill discharge will pass through the ball mill trommel prior to discharging to the cyclone feed pump box. Reject oversize material, from the ball mill trommel screen, will be collected within the ball mill scats bunker.

Grinding media (50 mm balls) will be added to the ball mill via the ball mill feed chute, utilising a dedicated media hoist, kibble and feed chute.

Spillage within the grinding circuit will be managed, utilising a dedicated drive-in sump and sump pump. Any spillage generated in the grinding area will be returned to the cyclone feed pump box.

Auxiliary equipment within the grinding area will include:

- SAG mill lubrication system.
- SAG mill liner handler, bolt removal tool and a relining monorail.
- SAG mill feed chute transporter for feed chute removal.
- SAG mill VSD.
- Ball mill lubrication system.
- Ball mill LRS and heat exchanger.
- SAG and ball mill feed chute removal winch.
- SAG mill ball charging hoist.
- Ball mill ball charging hoist.
- Cyclone maintenance jib crane.
- Metallurgical slurry sampler.

- Hydrogen Cyanide (HCN) gas monitor.
- Grinding circuit area cameras.

17.2.4 Pebble Crushing

Oversize from the SAG mill discharge screen will be conveyed to the pebble crusher feed bin, via a series of belt conveyors. A self cleaning belt magnet will be positioned at the head chute of the first conveyor to remove any scrap metal and steel media which could potentially damage the pebble crusher.

Downstream of the cross-belt magnet, the pebbles will pass under a metal detector, prior to discharging into the pebble crusher feed bin. The feed bin will provide surge capacity ahead of the pebble crusher and allow a controlled feed to be presented to the crusher. Should the pebble crusher not be operational, a diverter gate ahead of the pebble crusher feed bin will allow pebbles to bypass the pebble bin and crushers and feed directly to the pebble crusher discharge conveyor. Similarly, should the metal detector detect tramp metal (not removed by the cross-belt magnet), the diverter gate ahead of the pebble crusher feed bin will automatically allow pebbles to bypass the pebble bin and crushers and feed directly to the pebble crusher discharge conveyor.

Pebbles will be withdrawn from the pebble crusher feed bin, by a vibrating feeder, which will be variable speed. Initially, only a single pebble crusher will be installed, with provision made in the building design for future installation of a second pebble crusher. The abrasion index of the ore will require frequent replacement of the pebble crusher liner and will reduce grinding circuit throughput when the pebble liner change occurs. The pebble crusher will discharge crushed pebbles directly onto the pebble crusher discharge conveyor which in turn will return the crushed pebbles to the SAG mill feed conveyor.

The pebble crusher discharge conveyor will be fitted with a weightometer, used for mass accounting of feed presented to the grinding circuit.

Auxiliary equipment within the pebble crushing area will include:

- Pebble crusher lubrication unit and hydraulic power pack.
- Pebble crusher area cameras.

17.2.5 Leach Thickening

Trash screen undersize will flow by gravity directly to the leach thickener feed box, where flocculant will be added to aid with particle settling. Overflow solution from the leach thickener will flow by gravity to the leach thickener overflow tank and then be pumped to the carbon in columns circuit. Underflow from the leach thickener, at 45% solids, will be pumped by dedicated thickener underflow pumps, to the leach feed distribution box. A thickener recycle pump is included to improve thickener operational flexibility when running, or ensure compaction of the thickener bed does not occur if the thickener is off-line for a plant shutdown.

A cyanide analyser for on-line monitoring of the free cyanide concentration in the supernatant part of the leach thickener will allow for control of the sodium cyanide addition to the SAG mill.

The leach thickener area will be serviced by a dedicated floor sump pump. Spillage and wash down collected by the sump pump will be returned to the thickener feed box.

Auxiliary equipment within the leach thickener area will include:

- Leach thickener electric rake drive.
- Cyanide analyser.

17.2.6 Carbon in Columns Circuit

Leach thickener overflow will be pumped to the carbon in columns (CIC) circuit. This carbon adsorption circuit will recover gold in solution from the grinding circuit. The CIC circuit includes static feed and discharge screens that will prevent trash or particles from entering the circuit and carbon from leaving the circuit, respectively. The CIC circuit discharge, which is cyanide bearing solution, will be pumped to the process water tank for reuse in the grinding circuit.

A common carbon transfer pump will transfer carbon forward throughout the columns counter-current to the flow of solution as well as recover carbon to the loaded carbon recovery screen for carbon desorption. Approximately twice per week, loaded carbon from the first carbon column (or second carbon column if the first is off-line) will be pumped, by the carbon transfer pump, to the common loaded carbon recovery screen. The screen solution underflow will flow by gravity to the carbon column of origin whilst the loaded carbon will flow by gravity to the acid wash column.

Regenerated carbon (or fresh carbon) will be added to the CIC circuit, from the carbon regeneration circuit. The regenerated carbon (or fresh carbon) will be pumped, to the CIC circuit, via the CIC carbon sizing screen. The sizing screen will remove excess water and carbon fines. The dewatered carbon will discharge into the last, online, CIC tank with excess water and carbon fines directed to the carbon fines collection hopper for further removal from the circuit.

Process solution samplers will be fitted to the CIC feed and CIC discharge lines for gold metallurgical accounting purposes.

The CIC area will be serviced by a dedicated floor sump pump. Spillage and wash down collected by the sump pump will be returned to the carbon column feed box.

Auxiliary equipment within the CIC circuit area will include:

- Process solution samplers.
- Hydrogen Cyanide (HCN) gas monitor.

17.2.7 Leach Circuit

Leach thickener underflow will be pumped to the leach feed distribution box. The slurry from the leach feed distribution box will flow by gravity to the first leach tank. If the first leach tank is offline, the slurry will be diverted to the second leach tank, via a launder gate system.

The leach circuit will consist of five, mechanically agitated, leach tanks operating in series. This equates to a residence time of 24 hrs at a design feed rate of 4 Mtpa. Space allowance will be made for the inclusion of one additional leach tank for expansion of the plant throughput to 5 Mtpa and the 24 hr leach residence time is maintained. Each leach tank will have a live volume of 3,900 m³.

Oxygen generated on-site will be added, via oxygen addition devices, to the leach thickener underflow stream to ensure that all of the new feed slurry is mixed with oxygen before further sodium cyanide is added. Slurry pumps to recycle slurry and facilitate further oxygen addition have been included. The dissolved oxygen level target is 13 to 17 ppm for the leach circuit.

Lead nitrate will be added to the leach feed distribution box. Metallurgical testwork has shown improved gold leaching with the addition of this reagent.

Further sodium cyanide solution will be added to the first leach tank with a dedicated metering pump. Sodium cyanide can also be added to the third leach tank, if required, with its own dedicated metering pump. A cyanide analyser for on-line monitoring of the free cyanide concentration in the first and third leach tanks will allow for controlled addition of the sodium cyanide to the respective tank.

Process slurry samplers will be fitted to the leach tail stream for monitoring of leach circuit performance.

Should a leach tank be off-line for maintenance, it will be possible to bypass any of the leach tanks. The ability to bypass tanks will be made possible by the installation of pneumatic gates located within the leach inter-stage launders. One gate will divert slurry to the following leach tank while the second gate will allow slurry diversion to the subsequent leach tank.

The leach circuit will be serviced by three floor sump pumps. Sump pumps will return spillage to a nearby leach tank.

Auxiliary equipment within the leach circuit area will include:

- Process slurry sampler.
- Cyanide analyser.
- Hydrogen Cyanide (HCN) gas monitor.

17.2.8 Carbon in Pulp Circuit

The CIP circuit will consist of six, mechanically agitated, CIP tanks operating in series. This exceeds the required residence time of 6 hrs at design plant throughput of 4 Mtpa and meets the required residence time for a maximum plant throughput of 5 Mtpa. Each CIP tank will have a live volume of 1,100 m³.

The leaching circuit will dissolve the remaining gold in solid and the CIP circuit will recover this dissolved gold by carbon adsorption. Activated carbon will be retained in each of the CIP tanks by an inter-tank screen. The inter-tank screen will be a stainless steel wedge wire cylinder equipped with an internal agitator and external rotating wiper blade mechanism to prevent screen blinding and effectively pump the slurry to the next tank.

As the slurry flows by gravity through the CIP tanks, the carbon will be advanced counter-current to the slurry flow. This will be managed by balancing the carbon inventory in each CIP tank, by conducting regular measurements of the carbon concentration. Carbon advancement will be achieved by the CIP carbon transfer pumps (6 hrs per day), of which there is one transfer pump per CIP tank.

Approximately five times per week (3 hrs per batch), loaded carbon from the first CIP tank (or second CIP tank if the first is off-line) will be pumped, by the loaded carbon recovery pump, to the loaded carbon recovery screen, where it will be washed with spray water to remove excess slurry. The excess slurry (screen underflow) will flow by gravity to the CIP tank of origin whilst the loaded carbon will flow by gravity to the acid wash column.

Regenerated carbon (or fresh carbon) will be added to the CIP circuit, from the carbon regeneration circuit. The regenerated carbon (or fresh carbon) will be pumped, to the CIP circuit, via the CIP carbon sizing screen. The sizing screen will remove excess water and carbon fines. The dewatered carbon will discharge into the last, online, CIP tank with excess water and carbon fines directed to the carbon fines collection hopper for further removal from the circuit.

Slurry discharging the last CIP tank will flow by gravity to the CIP carbon safety screen via the carbon safety screen feed box. The carbon safety screen will capture and recover any carbon exiting the CIP circuit. The safety screen oversize will report to a fine carbon skip bin while the undersize will be pumped to the cyanide destruction feed box.

A cross stream metallurgical sampler, positioned prior to the cyanide destruction feed box will periodically collect a sample of the CIP tails stream. This sample will be used for circuit monitoring and for metal accounting.

Should a CIP tank be off-line for maintenance, it will be possible to bypass any of the CIP tanks. The ability to bypass tanks will be made possible by the installation of pneumatic gates located within the CIP inter-stage launders. One gate will divert slurry to the following CIP tank while the second gate will allow slurry diversion to the subsequent CIP tank.

The CIP circuit will be serviced by two floor sump pumps. Sump pumps will return spillage to either a nearby CIP tank or the CIP feed distribution box.

Auxiliary equipment within the CIP circuit area will include:

- CIP area gantry crane.
- Inter-stage screen maintenance frames, including a spare CIP inter-stage screen.
- High pressure washer for cleaning interstage screens.
- Metallurgical slurry sampler.
- Hydrogen Cyanide (HCN) gas monitor.

17.2.9 Acid Wash, Elution, Electrowinning and Gold Room

The Fekola desorption circuit will consist of separate acid wash and elution columns. A cold acid wash will be utilised for removal of inorganic foulants. Following acid wash, gold will be eluted from the carbon, utilising a Pressure Zadra elution process sized on the basis of completing a single strip cycle per 24 hr period. At a total carbon (gold) loading range of 2,100 to 6,500 g/t, the daily carbon movement of 12 tonnes will satisfy the required carbon movements for both the CIC and CIP circuits.

Acid Wash

The cold acid wash sequence will be required to remove accumulated, calcified scale, from the carbon surface. The acid wash column fill sequence will be initiated by pumping carbon, from the first CIP tank or first carbon column, into the acid wash column via the loaded carbon recovery screen. Carbon will flow by gravity from the loaded carbon recovery screen directly into the acid wash column. Once the Acid Wash Column is filled to the required level, the carbon fill sequence will be stopped.

The acid wash sequence will utilise a 3% w/v hydrochloric acid solution. This dilute acid will be prepared, by the addition of fresh water and neat (32%) hydrochloric acid, into the dilute acid tank. The acid wash sequence will involve the injection of the dilute acid solution into the column, by the acid wash circulation pump, via the feed manifold located beneath the column. The pump will continuously run and allow the acid to circulate through the column, back to the dilute acid tank. Acid circulation will occur for one hour or two bed volumes (2 BV).

Upon completion of acid circulation, the acid neutralisation sequence will commence by the addition of sodium hydroxide (caustic) sprayed into the top of the acid wash column. The acid wash circulation pump will continue to run and circulate the neutralised solution through the column and back to the dilute acid tank for one hour or two bed volumes (2 BV). Following completion of acid neutralisation, the rinse sequence will be initiated by pumping water through the column, to displace the neutralised acid solution to the tailings pump box. During the rinse step, two bed volumes (2 BV) of water, at 2 BV/hour, will be sourced from the treated water tank and pumped through the column.

This sequence will conclude with carbon being pumped to the elution column. Additional water to assist the carbon transfer between the acid wash and elution column will be sourced from the treated water tank.

Elution and Electrowinning

The elution sequence will commence with the addition of treated water into the barren strip solution tank along with the simultaneous injection of cyanide and caustic solution. A set amount of cyanide and sodium hydroxide (caustic) will be added to achieve a 2% w/w NaOH and 0.2% w/w NaCN solution. Both reagent additions will be automatically stopped once the prescribed volume has been added. The pre–heat period will then commence. During this period, the pre-heat solution will be circulated through the column and be pre–heated to 95°C via a diesel-fired hot water elution heater. Upon completion of the pre–heat period, the elution sequence will commence and gold will be stripped from the carbon in closed circuit with the electrowinning cells. The recovery heat exchangers will transfer residual heat from the pregnant strip solution exiting to the barren solution entering the elution column.

The pre-heated, incoming barren solution, will then pass through the primary heat exchanger to elevate the solution temperature to 145°C prior to entering the bottom of the column. A diesel-fired hot water elution heater will provide the heat to the primary heat exchanger. A temperature probe will monitor the temperature of barren solution entering the column and will be used to control the heater output. Hot solution will flow up and out of the top of the column, passing through the pregnant strip solution filters before the recovery heat exchangers, followed by the trim heat exchanger. The trim heat exchanger will use treated water to ensure the pregnant strip solution is reduced to the optimum temperature of 75°C for electrowinning. The electrowinning discharge will flow by gravity to the barren strip solution tank before being circulated through the elution and electrowinning circuits again. The elution and electrowinning sequence will occur for six hours or twelve bed volumes (12 BV), at 2 BV/hour.

Soluble gold, and any silver, will be recovered from the pregnant strip solution by electrowinning onto woven, stainless steel wire mesh cathodes. The electrowinning circuit will consist of three of four (duty/duty/duty/standby) electrowinning cells in a parallel configuration. Each electrowinning cell contains 33 cathodes and 36 anodes and has a dedicated rectifier to supply the necessary current, to electroplate the gold and silver onto the cathode.

During the elution sequence, the electrowinning cells will be in closed circuit with the elution column. The flow of pregnant solution to the cell will be evenly split across an electrowinning feed distributor and manual control valves will assist the desired linear velocity to be achieved. During this elution sequence, the electrowinning cell discharge will flow by gravity to the barren strip solution tank.

At the end of the elution sequence, the elution heater will be switched off, with 2 BV of treated water being utilised to cool down the elution column and its contents, to less than 100°C. Upon completion of the cool down sequence, the carbon will be pumped to the carbon regeneration kiln feed de-watering screen.

The acid wash area and elution area will be serviced by the acid wash area and elution area floor sump pumps respectively. Acid wash area spillage will be typically pumped to the tailings pump

box, with the option to be pumped to the carbon safety screen if there is carbon present in the area. The elution area spillage will be pumped to the CIP feed distribution box.

Auxiliary equipment within the acid wash, elution and electrowinning circuits will include:

- Elution heater expansion tank.
- Elution heater pump.
- Clean-in-Place system for the elution heater.
- Electrowinning cell extraction fans.

Gold Room

Upon completion of elution and electrowinning, gold sludge deposited on the cathodes will be washed off the cathodes, with a high pressure washer. The gold bearing sludge will be recovered to a sludge hopper, from where it will be filtered, via a sludge filter press.

The gold bearing filter cake will be thermally dried in an electric drying oven. The drying oven will be heated to approximately 400°C. Dried filter cake will be mixed with a prescribed flux mixture (silica, nitre, soda ash and borax), prior to being charged into the electric induction furnace. The fluxes added react with base metal oxides to form a slag, whilst the gold and silver remains as a molten metal. The molten metal will be poured into moulds, to form doré ingots, which will be cleaned, assayed, stamped and stored in a secure vault ready for dispatch. The slag produced will periodically be returned to the grinding circuit, via the SAG mill feed chute.

The gold room and electrowinning area will be serviced by a gold trap and dedicated gold room area sump pump. Any spillage within this area will be pumped back to the CIP distribution feed box.

Auxiliary equipment for the Electrowinning and Gold Room circuit will include:

- High pressure cathode washer.
- Smelting furnace dust collector and extraction fan.
- Smelting furnace cascade trolley and slag cart.
- Doré moulds and doré cleaning table.
- Flux bin, platform scale, flux mixing table.
- Doré balance.

17.2.10 Carbon Regeneration

After elution, the carbon will be pumped from the elution column to the carbon regeneration circuit. The carbon and transfer water will be directed to the kiln feed screen, allowing excess water to be removed prior to the carbon discharging into the kiln feed hopper. The kiln feed dewatering screen undersize will flow by gravity to the carbon fines collection hopper.

Carbon will be withdrawn from the kiln feed hopper, via the kiln screw feeder, and discharged directly into the carbon regeneration kiln, at a rate of 600 kg/h. Within the diesel-fired horizontal rotary kiln, the carbon will be heated to 700°C, to remove volatile organic foulants from the carbon surface, thereby restoring the carbon activity.

Re-activated carbon exiting the kiln will discharge directly to the carbon quench hopper, where it will be rapidly cooled. From the quench tank, carbon will be pumped, by the regenerated carbon transfer pump to either a carbon sizing screen located within the CIP circuit or a carbon sizing screen located within the CIC circuit. The carbon sizing screens undersize will flow by gravity to the carbon fines collection hopper.

Fresh new carbon, along with fresh water, will be added to the carbon conditioning hopper, which is mechanically agitated. After the carbon is conditioned for a period of 30 minutes, the conditioned carbon will be pumped, by the carbon addition transfer pump to either a carbon sizing screen located within the CIP circuit or a carbon sizing screen located within the CIC circuit. The new carbon sizing screens undersize will flow by gravity to the tailings pump box, via the carbon fines collection hopper.

The contents of the barren carbon fines collection hopper will be pumped by the carbon fines filter feed pump through the carbon fines filter to collect the carbon fines, which are subsequently removed from the circuit. The carbon fines filter requires a constant feed stream to ensure the filter cake doesn't slump within the press and ultimately cause a pressure differential across the plates that could result in filter press plate cracking. The filter will be operated until the filter is full of filtered cake. The filter press will be manually emptied and cleaned by the operator and subsequently restarted.

The carbon regeneration area will be serviced by dedicated floor sump pump. Any spillage generated within this area will be typically pumped to the carbon sizing screen or tailings pump box with an option to be pumped to the CIP tails pump box.

17.2.11 Cyanide Destruction

CIP tailings will be pumped to the cyanide destruction tank where cyanide destruction will be achieved utilising the SO_2 / Air process. In the SO_2/Air process, sodium metabisulphite, air, copper sulphate (catalyst) and sodium hydroxide (caustic) will be added to oxidise the residual free and WAD (weak acid dissociable) cyanide to cyanate. The destruction circuit will reduce the residual cyanide contained within the tailings stream to below 10 mg/L WAD cyanide. The cyanide destruction circuit will consist of two mechanically agitated tanks, operating in parallel, to provide a total residence time of 90 minutes.

Slurry exiting the cyanide destruction circuit will be pumped to the high density tailings thickener, via a process sampler. This sample will be used for circuit monitoring and CN_{wad} determinations of the slurry. The CN_{wad} levels of the cyanide destruction discharge will be measured by the on-site laboratory.

The cyanide destruction circuit will be serviced by a dedicated floor sump pump. Any spillage within this area will be returned to the cyanide destruction feed box.

17.2.12 Tailings Thickening and Disposal

Slurry from the cyanide destruction circuit will be pumped, via a process sampler, to the tails thickener feed box. Flocculant will be added to the tails thickener to enhance the settling properties of the solids. Overflow from the tails thickener will flow by gravity to the reclaim water pond.

Tailings thickener underflow, at a slurry density of 60% solids, will be pumped to the tailings pump box. Two tailings pumps, in series configuration, will pump to the tailings storage facility (TSF) and discharge the thickened slurry via spigots around the circumference of the dam. Water from the surface of the TSF will be recovered from the decant system and pumped directly to the reclaim water pond at the process plant. Underdrainage and seepage from around the TSF drainage system will be pumped into the TSF for recovery by the decant return water pump.

The tailings thickening circuit will be serviced by one floor sump pump. Any spillage collected within this area will be directed to the tailings thickener feed box.

Auxiliary equipment for the tailings thickening circuit will include:

- Tailings thickener electric rake drive.
- Process slurry sampler.

17.2.13 Reagents Mixing and Storage

The major reagents utilized within the process plant will include:

- Quicklime (CaO) for pH control.
- Sodium Cyanide (NaCN) for gold dissolution and desorption.
- Lead Nitrate ($Pb(NO_3)_2$) for enhancing gold dissolution.
- Caustic Soda (NaOH) for carbon acid washing neutralisation and desorption.
- Hydrochloric Acid (HCl) for carbon acid washing.
- Sodium Metabisulphite (SMBS) for cyanide destruction.
- Copper Sulphate Pentahydrate ($CuSO_4.5H_2O$) for cyanide destruction.

- Flocculant for thickening.
- Antiscalant to minimise scaling in the process water distribution, reclaim water distribution, fresh water distribution, gland water distribution, and elution circuit.
- Fluxes for smelting.

Quicklime

Quicklime will be delivered to site in 1,200 kg bulk bags. The bulk bag will be lifted, by the lime hoist, to the bulk bag splitter mounted above the lime transfer hopper. The lime will be pneumatically transferred to the lime silo. The lime storage silo will have a storage capacity of three days at design consumption rates at 4 Mtpa.

Quicklime will be withdrawn from the lime silo, by a rotary valve feeder and discharged directly to the SAG mill feed conveyor.

Sodium Cyanide (NaCN)

Sodium cyanide will be delivered to site in 1,000 kg bulk bags in boxes. The bulk bag will be lifted, by the cyanide hoist, to the bulk bag splitter mounted above the cyanide mixing tank.

Solid cyanide briquettes will be released, from the bulk bag, by the bag splitter. Fresh water will be added to the mixing tank to achieve a solution with the desired cyanide concentration (20% w/v). The mixing tank will be mechanically agitated to assist with cyanide dissolution.

Sodium hydroxide will be added to the cyanide mixing tank, to maintain the pH above 12, during the mixing process. Control of the pH above 12 minimises formation of gaseous hydrogen cyanide (HCN) during the mixing process.

Following completion of the mixing cycle, sodium cyanide solution will be transferred to the cyanide storage tank, from which dedicated metering pumps will deliver cyanide solution to the SAG mill feed chute, leach tanks and elution circuit.

The sodium cyanide and sodium hydroxide mixing and storage areas will be serviced by a common floor sump pump. Any spillage generated within this area will be pumped to the first leach tank.

Lead Nitrate ($Pb(NO_3)_2$)

Lead nitrate will be delivered to site in 1,000 kg bulk bags. The bulk bag will be lifted, by the lead nitrate hoist, to the bulk bag splitter mounted above the lead nitrate mixing tank.

Solid lead nitrate will be released, from the bulk bag, by the bag splitter. Fresh water will be added to the mixing tank to achieve a solution with the desired lead nitrate concentration (20% w/v). The mixing tank will be mechanically agitated to assist with lead nitrate dissolution.

Following completion of the mixing cycle, lead nitrate solution will be transferred to the lead nitrate storage tank, from which dedicated metering pumps will deliver lead nitrate solution to the leach feed distribution box.

The lead nitrate mixing and storage area will be serviced by a dedicated floor sump pump. Any spillage generated within this area will be pumped to the leach feed distribution box.

Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda – 'caustic') will be delivered to site in 1,200 kg bulk bags. The bulk bag will be lifted, by the caustic hoist, to the bulk bag splitter mounted above the caustic mixing tank.

Sodium hydroxide will be released, from the bulk bag, by the bag splitter. Fresh water will be added to the mixing tank to achieve a solution with the desired caustic concentration (20% w/v). The mixing tank will be mechanically agitated to assist with caustic dissolution. Following completion of the mixing cycle, sodium hydroxide solution will be transferred to the caustic storage tank, from which dedicated metering pumps will deliver caustic solution to the acid wash column, elution circuit, cyanide mixing tank and the cyanide destruction tank, if required.

Hydrochloric Acid (HCl)

Hydrochloric acid (32% w/w) will be delivered to site in 1,000 L totes or bulk containers (IBC). The acid transfer standpipe and pump will transfer acid, from the bulk containers, to the dilute acid tank. Fresh water will be added to the dilute acid tank to achieve a solution with the desired acid concentration (3% w/w). The dilute acid tank will be mixed by using the acid wash circulation pumps. Following completion of the mixing cycle, the dilute acid pumped to the acid wash column during the acid wash sequence.

The hydrochloric acid storage area will be serviced by an air operated dedicated floor sump pump. Any spillage generated within this area will be pumped to the tailings pump box.

Sodium Metabisulphite (SMBS)

Sodium metabisulphite will be delivered to site in 1,200 kg bulk bags. The bulk bag will be lifted, by the SMBS hoist, to the bulk bag splitter mounted above the SMBS mixing tank.

SMBS will be released, from the bulk bag, by the bag splitter. Fresh water will be added to the mixing tank to achieve a solution with the desired SMBS concentration (20% w/v). The mixing tank will be mechanically agitated to assist with SMBS dissolution. Following completion of the mixing cycle, SMBS solution will be transferred to the SMBS storage tank, from which the dedicated metering pumps will deliver SMBS solution to the cyanide destruction circuit, as the source of sulphur dioxide (SO_2) required for the oxidation of cyanide.

The SMBS and $CuSO_4$ mixing and storage area will be serviced by a common floor sump pump. Any spillage generated within this area will be pumped to the cyanide destruction feed box.

Copper Sulphate Pentahydrate (CuSO4.5H2O)

Copper sulphate will be delivered to site in 1,000 kg bulk bags. The bulk bag will be lifted, by the hoist, to the bulk bag splitter mounted above the copper sulphate mixing tank.

Fresh water will be added to the mixing tank to achieve a solution with the desired $CuSO_4$ concentration (15% w/v). The mixing tank will be mechanically agitated to assist with $CuSO_4$ dissolution. Following completion of the mixing cycle, $CuSO_4$ solution will be transferred to the CuSO4 storage tank, from which the dedicated metering pumps will deliver $CuSO_4$ solution to the cyanide destruction circuit, as the catalyst for the cyanide oxidation and destruction.

Flocculant

Flocculant micro-breads powder will be delivered to site in 700 kg bags. The bulk bag will be lifted, by the flocculant hoist, to the storage hopper. Flocculant will be mixed in a proprietary mixing system, comprised of a bulk dry hopper, screw feeder, flocculant blower and mixing tank. The flocculant plant will mix flocculant powder with fresh water to achieve the required storage concentration (0.25% w/w).

Flocculant will be withdrawn from the storage hopper by the flocculant screw feeder. The screw feeder will convey flocculant to the flocculant eductor, from which the flocculant powder will be pneumatically conveyed, to the flocculant mixer, by the flocculant blower. Fresh water will be added to the mixer, to hydrate the flocculant powder, prior to discharging into the agitated flocculant mixing tank. Upon completion of the mixing cycle, the flocculant will be transferred to the flocculant storage tank, by the flocculant transfer pump.

The process plant layout means that there will be two flocculant mixing plants, one adjacent to the leach thickener, and the other next to the tailings thickener. Flocculant will be distributed to either the leach thickener or the tailings thickeners (via in-line static mixers) from the respective storage tank by dedicated metering pumps.

Each flocculant area will be serviced by a dedicated floor sump pump. Any spillage generated within this area will be pumped to the tailings pump box.

Antiscalant

Antiscalant will be delivered to the plant in totes or bulk containers (IBC). Metering pumps will distribute antiscalant, directly from the IBC, to the process water, reclaim water, fresh water, gland water, and elution circuits.

Fluxes

The following fluxes, will be delivered to the plant in 25 kg bags, and used in the gold room; borax, $Na_2B_4O_2$, sodium nitrate (Nitre), $NaNO_3$, soda ash (sodium carbonate), Na_2CO_3, and silica sand (SiO_2).

17.2.14 Water Services

The process plant will utilise process water, reclaim water, fresh water, treated water, gland water and potable water. Any shortfall of process water will be made up, preferentially, from water contained within the reclaim water pond, which includes water from the tailings decant. If insufficient water is available within the reclaim water pond, fresh water will be utilised for filling up the reclaim water pond, as required. An event pond, which will hold any overflow from the process plant and stormwater collected from around the process plant, will be pumped to the reclaim pond when necessary.

Process Water

Process water will predominantly consist of leach thickener overflow, via the carbon columns, and reclaim water make-up. From the process water tank, process water will be reticulated throughout the plant via dedicated process water pumps, with off-takes supplied for the predominant user points, namely:

- SAG mill feed chute.
- Cyclone feed pump box.

Reclaim Water

Reclaim water will predominantly consist of tailings thickener overflow, decant return water from the TSF and fresh water overflow from the fresh water tank. From the reclaim water pond, reclaim water will be reticulated throughout the plant via dedicated reclaim water pumps, with off-takes supplied for the predominant user points, namely:

- Process water tank.
- Flocculant dilution water.
- Cyanide destruction feed dilution water.
- Numerous screen water sprays and.
- Service points around the plant.

Fresh Water and Fire Water

Fresh water for the process plant and mining operation will be sourced from active pit dewatering bores. The location of the pit dewatering bores will change as the mining progresses through the stages of the mine life. The bores will pump predominantly to the fresh water tank, and if required, the bores will pump to the fresh water storage pond. Under normal conditions, the fresh water tank will be full and overflow to the reclaim water pond, which is used to make-up the process water demand.

The fresh water tank will provide a combined fresh water and fire water reserve. Fresh water from this tank will be reticulated throughout the plant via dedicated fresh water pumps to the predominant user points, namely:

- Primary crusher circuit dust suppression.
- Grinding circuit cooling water.
- Numerous screen water sprays.
- Carbon regeneration.
- Reagent mixing.
- Water treatment plant supply for the elution circuit.
- Gland water supply.

From the fresh water tank, fresh water will be distributed to the grinding circuit cooling water system. This is a closed circuit cooling water system, with cooling water being returned to the fresh water tank. In the elution circuit, the trim heat exchanger will use treated water to control the final pregnant solution temperature before electrowinning. The cooling water from the trim heat exchanger will be returned to the fresh water tank.

Firewater will be supplied from the plant fresh water storage tank, via a dedicated suction manifold. The firewater system will comprise:

- An electrical jockey pump.
- An electrical firewater pump.
- A diesel standby firewater pump.

The firewater system pressure will be maintained, by the jockey water pump. An electric fire water pump will automatically start on a drop in line pressure. The diesel fire water pump will automatically start if the line pressure continues to drop below the target supply pressure, which will occur when there is significant fire water demand or during a power failure.

Gland Water

Gland water will be filtered and stored in a dedicated gland water storage tank. Dedicated gland water pumps will supply gland seal water to the required pumps across the process plant.

Potable Water

Fresh water for potable water use will be sourced from dedicated potable water bores. The location of the potable water bores will be relatively close to the camp and process plant. The bores will pump to the potable water feed tank. Potable water will be generated by a treatment

plant with filtration and reverse osmosis (RO) to reduce the total dissolved solids content. Potable water, from the treatment plant, will be stored within a dedicated potable water tank before being sterilised with ultra-violet light and distributed for use. The backwash stream from the treatment plant will be pumped to the tailings pump box.

Potable water will be distributed for human consumption across the site and to the safety showers and eye wash stations throughout the process plant. The safety showers are reticulated from a dedicated safety shower water tank via a ring-main and dedicated safety shower water pumps. If the water temperature in the safety shower water tank becomes too high, then the ring-main will be discharged to the fresh water tank whilst the water in the safety shower water tank is refreshed with replacement potable water.

17.2.15 Air Services

High Pressure Air

High pressure air at 700 kPag will be provided by two high pressure air compressors, operating in a lead-lag configuration. The entire high pressure air supply will be dried and can be used to satisfy both plant air and instrument air demand. Dried air will be distributed via one dedicated air receiver for the entire process plant.

Low Pressure Air

Low pressure (LP) air will be supplied by one of two, low pressure air blowers. Low pressure air will be reticulated to the cyanide destruction circuit.

Oxygen Plant

Oxygen, for use within the Leach circuit, will be supplied by one of two, oxygen Vacuum Swing Adsorption (VSA) plants. Oxygen will be generated to a supply pressure of 700 kPag. Oxygen will be distributed to the required leach circuit area.

17.3 Plant Consumption

17.3.1 Energy Consumption

The power demand for the processing plant, along with the rest of the site and camp, will be provided by on-site power generation using heavy fuel oil (HFO) and diesel fuel. The average power demand is summarised in Table 17.3.1 and utilised for the operating cost estimate. The average power demand does not reflect the instantaneous power demand for equipment start-up and power plant capacity sizing.

Table 17.3.1 Average Power Demand Summary

Plant Areas	Installed Power (kW)	Average Continuous Draw (kW)	Total Annual Power Consumption (kW / year)
120 - Feed Preparation	825	348	3,044,793
130 - Milling	25,722	16,371	143,413,935
140 - Trash Removal / Thickening	622	315	2,756,322
160 - Leaching	2,484	1,551	13,587,329
170 - Elution / Goldroom	556	113	988,649
180 - Tails Handling	1,905	905	7,930,650
210 - Reagents	238	65	567,973
220 - Water Services	1,415	601	5,268,175
230 - Plant Services	68	38	331,004
240 - Air Services	1,495	663	5,806,478
450 - Mining Facilities	66	39	344,315
Total	**35,395**	**21,009**	**184,039,625**

17.3.2 Water Consumption

A water balance for the process plant has been completed. Water from the leach thickener and tailings thickener overflow streams are recycled within the process plant to reduce the external water requirement. Approximately 200 m^3/h of decant return water is recycled from the TSF to the process plant. Another approximately 200 m^3/h of fresh water is required to make-up the water consumption for the process plant. The fresh water will primarily be sourced from nearby pit dewatering bores. Additional water will be stored in the Fresh Water Storage Pond and is available for use in the plant.

At the process plant, the fresh water tank will provide a combined fresh water and fire water reserve. The required fire water capacity will be stored below the level of the suction piping for the fresh water pumps to ensure it is always available.

17.3.3 Reagent and Consumable Consumption

Reagent storage, mixing and pumping facilities will be provided for all reagents for the process plant. Table 17.3.2 provides a summary of reagents and consumables that will be used at the process plant at the design consumption rate for a plant throughput of 4 Mtpa.

Table 17.3.2 Annual Reagent and Consumable Consumption

Reagent / Consumable	Annual Consumption
Gyratory Crusher Liners	1.5 sets
Pebble Crusher Liners	14.0 sets
SAG Mill Liners	1.9 sets
Ball Mill Liners	2.5 sets
SAG Mill Grinding Media	760 t
Ball Mill Grinding Media	4,240 t
Quicklime	2,960 t
Sodium Cyanide	3,124 t
Lead Nitrate	400 t
Activated Carbon	120 t
Sodium Hydroxide (Caustic)	160 t
Hydrochloric Acid	529 t
Sodium Metabisulphite	3,922 t
Copper Sulphate	236 t
Flocculant	248 t
Borax	4.5 t
Sodium Nitrate (Nitre)	0.3 t
Sodium Carbonate (Soda Ash)	0.3 t
Silica Sand	1.7 t
Smelting Furnace Crucibles	11 each
Diesel Fuel	2,119 kL
Antiscalant	99 t

17.4 Plant Control System

The instrumentation and control system will be designed with the following objectives in mind:

- Ease of operation - operator intervention and remedial action will be easily accomplished.
- Interlocking - the control system will provide process interlocking between individual drives and plant areas.
- Reliability - all field equipment will be selected on its ability to operate in an environment where extreme temperatures and dusty conditions are encountered.
- Ease of maintenance - instrumentation and control systems will be designed and installed such that maintenance requirements are minimised.

The plant control system will be designed around a moderate level of automation and monitoring.

The starting and stopping of most electrical drives and actuated valves will be performed from the Distributed Control System (DCS) Operator Interface Terminals (OITs). There will be no Remote / Local selector switches in the field. However, field operators may start equipment local to the drive

if the control room has selected *Local* at the OIT. Local mushroom head style Lock-Off-Stop stations will be installed which will be hard-wired to the drive starter circuit to enable local stop / emergency stop regardless of drive mode selection.

In general, the use of actuated isolation or control valves will be implemented around the plant for automatic control loops or sequencing as part of the plant control or the elution sequence. All actuated valves and control valves will be operated from the OITs with remote position indication available. Automatic control valves will be controlled by PID loops within the DCS.

Equipment in the plant will be interfaced to the DCS via the Input / Output (I/O) modules, which are positioned in the field and remote to the main DCS hardware. The hardware will be housed in an office next to the main mill control room. Vendor equipment packages throughout the plant will utilise the process plant DCS for control and monitoring of their respective equipment.

All fixed speed drives are controlled using Profibus connected motor protection relays. The current draw of the majority of fixed speed drives will be displayed on a number of summary mimic pages on the DCS and adjacent to the drive mimic where appropriate. Similarly, the current draw of all variable speed drives will be displayed on a number of summary mimic pages on the DCS and adjacent to the drive mimic where appropriate.

The DCS will perform all digital and analogue control functions, including PID control. Faceplates on the DCS displays will facilitate the entry of setpoints, readout of process variables (PV's) and controlled variables (CV's) and entry of the three PID parameters (Proportional, Integral and Derivative).

The majority of equipment interlocks will be software configurable. However, selected drives will be hard wired to provide the required level of personal safety protection e.g. the emergency stop buttons associated with each and every motor and the pull wire switches associated with conveyors.

All alarm and trip circuits from field or local panel mounted contacts will be based on fail-safe activation. Alarm and trip contacts will open on abnormal or fault condition. If equipment shutdown occurs due to loss of mains power supply, the equipment will return to a de-energised state and will not automatically restart upon restoration of power.

Sequential group starts and sequential group stops will not be incorporated with non-packaged plant equipment. However, in any sequential process, critical safety and equipment protection interlocks will cause a cascade stop in the event of interlocked downstream equipment stopping (e.g. trip of mill feed conveyor will result in stop of stockpile reclaim apron feeder).

The process plant will be provided with a one main control room.

Operator Interface Terminals (OIT) will be provided in the following areas:

- DCS hardware office (1 engineering OIT).
- Main plant control room (2 OITs).

- Crusher control room (1 OIT).
- Elution circuit area (1 OIT).

FEKOLA GOLD PROJECT

NI 43-101 - PROJECT INFRASTRUCTURE

3174-STY-002

Table of Contents

			Page
18.0	**PROJECT INFRASTRUCTURE**		**18.1**
	18.1	Overall Site Development	18.1
	18.2	Roads	18.3
		18.2.1 Access to Site	18.3
		18.2.2 Project Site Roads	18.3
		18.2.3 Access Tracks	18.3
	18.3	Airstrip	18.3
	18.4	Tailings Storage Facility (TSF)	18.4
		18.4.1 Capacity and Location	18.4
		18.4.2 Design Considerations	18.4
		18.4.3 Design Summary	18.4
		18.4.4 Monitoring	18.11
		18.4.5 Rehabilitation	18.11
		18.4.6 Tailings Geochemistry	18.11
		18.4.7 Geotechnical	18.13
		18.4.8 Seepage Assessment	18.13
		18.4.9 Stability Analysis	18.14
		18.4.10 Decant from Tailings Storage Facility	18.15
	18.5	Surface Water Management	18.15
		18.5.1 Design Objectives	18.15
		18.5.2 Diversion Structures	18.16
		18.5.3 Collection and Control Structures	18.16
		18.5.4 Contact Water Storage Pond	18.19
		18.5.5 Sediment Control Structures	18.19
		18.5.6 Falémé River Flood Assessment	18.19
	18.6	Water Supply	18.21
		18.6.1 Pit Groundwater	18.21
		18.6.2 Surface Water	18.22
		18.6.3 Falémé River Bores	18.22
		18.6.4 Water Balance	18.22
	18.7	Power	18.23
		18.7.1 Power Generation	18.23
		18.7.2 Electrical Distribution	18.23
		18.7.3 Electrical Buildings	18.24
		18.7.4 Transformers and Compounds	18.24
	18.8	Fuel Supply	18.25
	18.9	Potable Water	18.25
		18.9.2 Process Plant	18.25
		18.9.3 Accommodation Camp	18.25
	18.10	Sewage and Solid Waste Management	18.26
		18.10.1 Sewage Treatment	18.26
		18.10.2 Solid Wastes	18.26
	18.11	Accommodation Camp	18.26
	18.12	Plant Facilities	18.27
		18.12.1 General	18.27
		18.12.2 Mine Services Area Facilities	18.27
		18.12.3 Process Plant Area	18.28
		18.12.4 Other Support Facilities	18.28

Table of Contents (Continued)

Page

TABLES

Table 18.4.1	TSF Stage / Storage Capacity	18.9
Table 18.4.2	Desired Minimum Factors of Safety	18.14
Table 18.4.3	Summary of TSF Embankment Stability Analysis Results	18.14
Table 18.4.4	Average conditions – TSF recycle rates	18.15
Table 18.9.1	Potable Water Demand	18.25

FIGURES

Figure 18.1.1	Overall Site Development Plan	18.2
Figure 18.4.1	TSF General Arrangement Stage 1	18.5
Figure 18.4.2	TSF General Arrangement Final Stage	18.6
Figure 18.4.3	TSF Typical Embankment Sections	18.7
Figure 18.4.4	TSF Typical Embankment Sections	18.8
Figure 18.5.1	Fresh and Contact Water Storage Ponds - General Arrangement	18.17
Figure 18.5.2	Fresh and Contact Water Storage Ponds - Typical Embankment Sections	18.18

18.0 PROJECT INFRASTRUCTURE

18.1 Overall Site Development

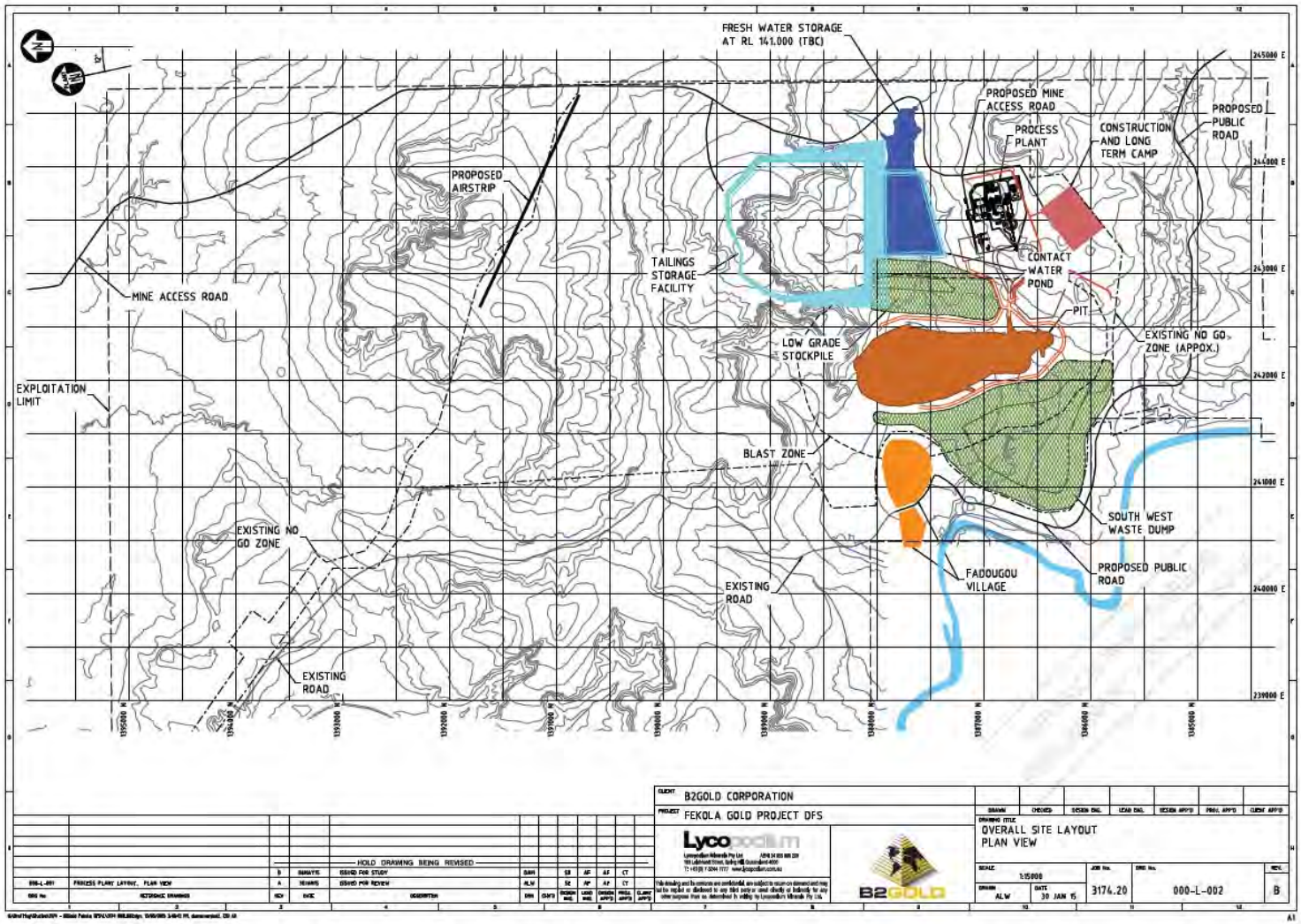
The overall site development plan is shown in Figure 18.1.1. The drawing shows the major features of the Project and its infrastructure including the process plant, tailings storage facility, accommodation camp, roads, airstrip, mine services area, mine open pit and mine waste dumps.

The process plant and tailings storage facility are located on the eastern side of the Fekola open pit, with all work areas outside of the 500 m blast zone. The accommodation camp is located close to the southern perimeter of the process plant for ease of personnel access.

The main access road approaches the site from the north and runs adjacent to the airstrip and tailings storage facility before turning west to access the process plant.

The site as a whole will be fenced to clearly delineate the mine area, prevent animal access and deter access by unauthorised persons. Road access into the fenced area will be through a manned checkpoint. Security fencing will surround the accommodation camp and general site infrastructure. High security fencing will surround the process plant.

Figure 18.1.1 Overall Site Development Plan



18.2 Roads

18.2.1 Access to Site

Materials and consumables will be transported to site via a new mine access road. The new access road will link the project site with the existing Millennium Highway approximately 20 km from the nearby town of Kéniéba and will be a six metre wide gravel, all weather, and free draining carriageway. Culverts will be installed as appropriate on creek crossings. Fill material for the road will be obtained from excavation alongside the road where possible.

The new road has been routed to avoid the villages of Brema, Gonu, Kéniéba, Mantakoto, Médinandi, Pandola, Sanougou and Satadougou Tintiba and cultural points identified at these villages. The main access road will be designed for a 60 km/h speed limit.

Construction of the access road commenced as part of project early works and is expected to be fully completed prior to the 2015 wet season to enable continued all weather access to site for construction.

18.2.2 Project Site Roads

Plant internal roads will provide access between the administration area, process plant facilities, fuel storage, power plant, mine services area, and accommodation camp. These roads will generally be 6 m wide and will be constructed flush with bulk earthworks pads to ensure that storm water sheet flow is achieved across the site, thereby avoiding the need for deep surface drains and culvert crossings within the plant area.

The airstrip is located adjacent to the mine access road to the north of the tailings storage facility. A suitable road will connect the airstrip to the mine access road to provide access for the transport of personnel from the airstrip to the camp.

18.2.3 Access Tracks

A number of new tracks will be constructed to access infrastructure such as the tailings storage facility, sediment control structures and water bore pumps remote from the plant site. The access tracks will be cleared and graded natural earth tracks. Exact routes will be determined during construction of the Project to best fit local terrain and vegetation density.

18.3 Airstrip

A 1,800 m long (1,600 m active length) x 30 m wide all weather gravel airstrip will be provided for secure transport of bullion, transportation of construction personnel, and emergency medivac purposes. The airstrip has been designed to suit a Beech Craft 1900 type aircraft or similar.

The airstrip will have an orientation bearing 114 deg to suit prevailing wind conditions and include clearance zones of 100 m at the ends and 25 m on the sides. An area of 2,600 m² will be reserved for aircraft parking and include concrete anchorages to secure steel cables to the aircraft in case of strong winds.

No allowance has been made for lighting for night landings or takeoffs.

18.4 Tailings Storage Facility (TSF)

18.4.1 Capacity and Location

The TSF is located in the valley to the north of the Plant Site and Pit, adjacent to the eastern waste dump. The TSF was designed to store a total of 62 Mt of tailings with a throughput of 5 Mtpa. Stage 1 has been design for a 16 month period to store 6.7 Mt of tailings to allow for flexibility for the Stage 2 construction to be carried out during the subsequent dry season.

18.4.2 Design Considerations

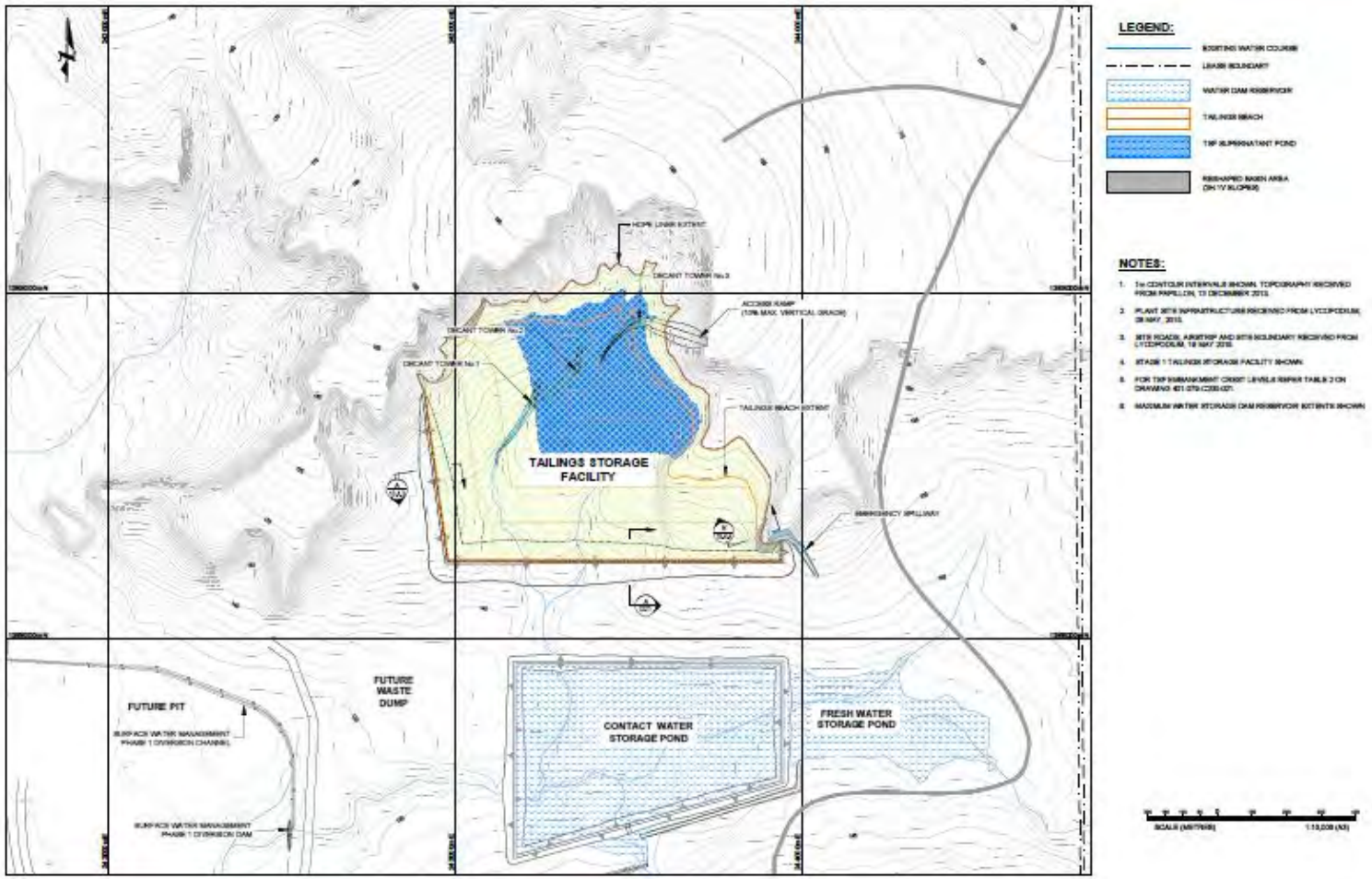
The design objectives for the TSF are as follows:

- Permanent and secure containment of all solid waste materials.
- Maximisation of tailings densities using sub-aerial deposition.
- Removal and reuse of free water.
- Reduction of seepage.
- Containment of extreme storm events within the TSF.
- Ease of operation.
- Rapid and effective rehabilitation.

18.4.3 Design Summary

The TSF will comprise a valley storage formed by a single downstream multi-zoned earthfill embankment, lined with high density polyethylene (HDPE), comprising a total footprint area (including the basin area) of approximately 69 ha for the Stage 1 TSF, increasing to 200 ha for the final TSF. The TSF is designed to store a total of 62 Mt of tailings. The Stage 1 TSF was designed to provide 16 months of storage capacity, in order to allow some flexibility for Stage 2 construction to be carried out during the subsequent dry season, when conditions are more suitable for construction. The TSF general arrangement for Stage 1, Final Stage 12 and typical embankment sections are shown in the following Figures 18.4.1, 18.4.2, 18.4.3 and 18.4.4.

Figure 18.4.1 TSF General Arrangement Stage 1



Knight Piésold CONSULTING

FEKOLA GOLD PROJECT
TAILINGS STORAGE FACILITY
GENERAL ARRANGEMENT STAGE 1

Ref: NI 43-101
Figure 18.4.1

Figure 18.4.2 TSF General Arrangement Final Stage

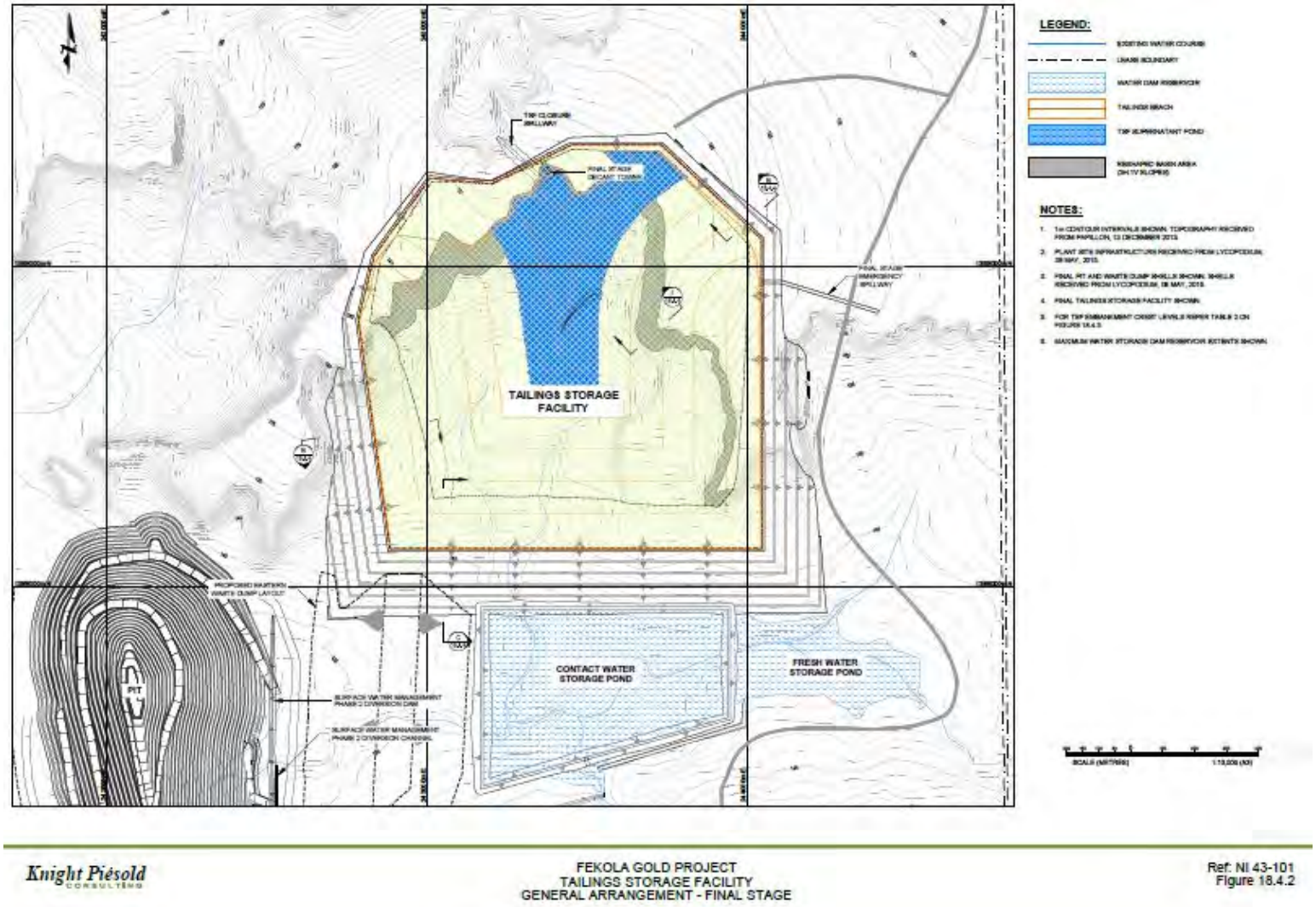


Figure 18.4.3 TSF Typical Embankment Sections

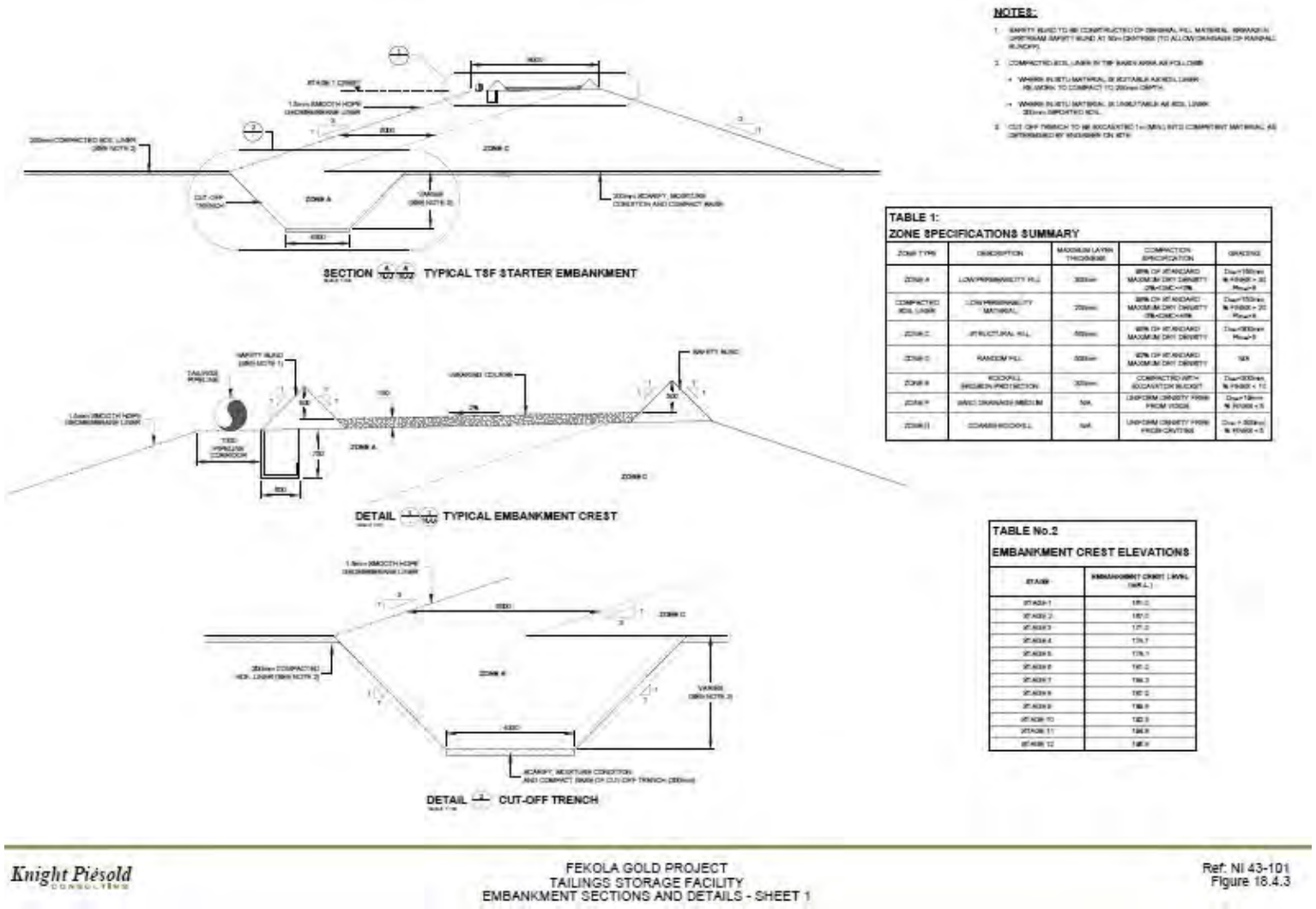
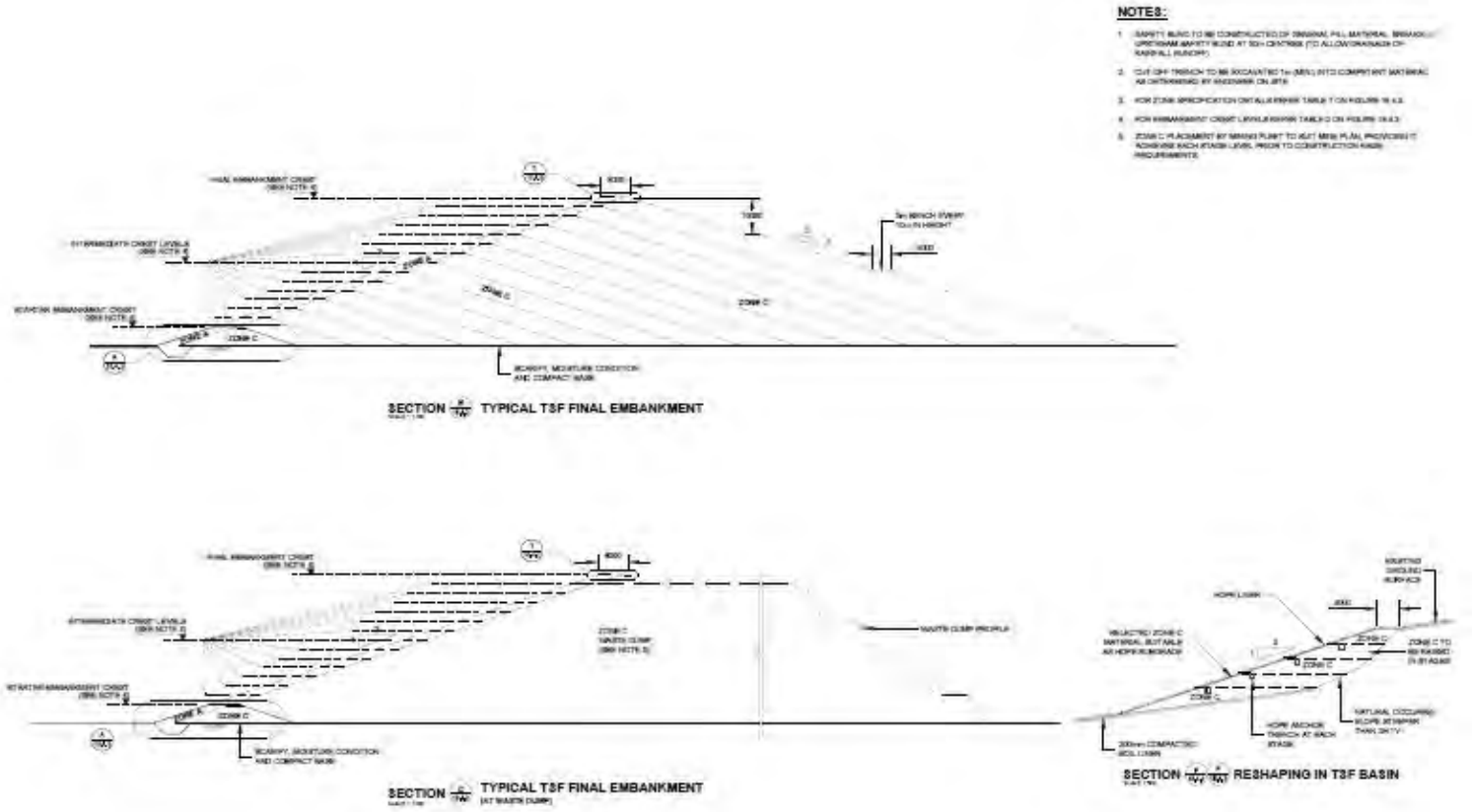


Figure 18.4.4 TSF Typical Embankment Sections



Knight Piésold CONSULTING

FEKOLA GOLD PROJECT
TAILINGS STORAGE FACILITY
EMBANKMENT SECTIONS AND DETAILS - SHEET 2

Ref: NI 43-101
Figure 18.4.4

The TSF embankments will be constructed in annual raises to suit storage requirements. Downstream raise construction methods will be utilised throughout operation. Staged embankment crest elevations are presented in Table 18.4.1.

Table 18.4.1 TSF Stage / Storage Capacity

Stage	Tailings Storage (Cumulative) (Mt)	TSF Embankment Elevation[*1] (m RL)	Maximum TSF Embankment Height (m)
1[*2]	6.7	161.0	22.0
2	11.7	167.0	28.0
3	16.7	171.0	32.0
4	21.7	174.7	35.7
5	26.7	178.1	39.1
6	31.7	181.2	42.2
7	36.7	184.3	45.3
8	41.7	187.2	48.2
9	46.7	189.8	50.8
10	51.7	192.3	53.3
11	56.7	194.8	55.8
12	62.0	196.8	57.8

***1 Includes a minimum freeboard and stormwater capacity for the greater of: (i) 1 in 100 year recurrence interval, 72 hour storm event occurring on an average conditions pond, or (ii) 1 in 100 year wet rainfall sequence pond.**

***2 Stage 1 embankment designed for 16 month storage capacity.**

The embankment comprises an upstream low permeability zone (Zone A) and downstream structural fill zone (Zone C). The embankment upstream face will be lined with HDPE liner. Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow to form the low permeability zone of the embankment.
- Zone C material shall be won from borrow or delivered by the mining operations to form the structural fill zone of the embankment.

The embankment will have an operating upstream and downstream slope of 3H:1V and a minimum crest width of 8 m. The final downstream embankment profile will consist of an overall slope of approximately 3.5H:1V.

The TSF basin area will be cleared, grubbed and topsoil stripped. The upper slopes within the basin will be reshaped with imported materials, where required, to a maximum allowable slope of 3H:1V in order to achieve suitable slopes for HDPE liner installation.

A 200 mm depth compacted soil liner, will be constructed over the entire TSF basin area, comprising either reworked in-situ material or imported Zone A material, to form a subgrade for HDPE liner. A 1.5 mm HDPE geomembrane liner will be installed over the entire TSF basin and upstream embankment faces.

A cut-off trench will be excavated into foundation soils to competent foundation material, backfilled with low permeability (Zone A) fill to reduce seepage losses through the embankment foundations. The cut-off trench (estimated average 3.5 m depth) will be cut to extend through to competent low permeability foundation material.

The TSF design incorporates a basin underdrainage system to reduce pressure head acting on the geomembrane liner, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments. The underdrainage system comprises a network of branch drains reporting to collector drains situated in natural drainage courses. Underdrains will comprise perforated plastic pipes, covered in sand / gravel drainage material and wrapped in geotextile. The underdrainage system drains by gravity to a collection sump located at the lowest point in the TSF basin. Solution recovered from the underdrainage system will be released to the top of the tailings mass via submersible pump, reporting to the supernatant pond.

Supernatant water will be removed from the TSF via submersible pumps located within decant towers, constructed at start-up and raised during operation. Each decant tower will be situated in a sump within the inlet trench to ensure sufficient depth of water prior to pump operation within the tower. Each decant tower will comprise a 25 MPa concrete base and 1.8 m diameter slotted precast concrete pipe sections. The decant tower will be surrounded by free-draining coarse rockfill (Zone G). A pump bipod stand will be installed over the decant tower to facilitate pump removal. The supernatant pond will be maintained on the northern edge of the TSF basin. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit.

The TSF has sufficient capacity to completely contain all design criteria storm events and rainfall sequences (average recurrence interval of 1 in 100 years). Under normal operating conditions, with the TSF managed in accordance with standard operating procedures, the available stormwater storage capacity will be in excess of the design storm event volumes and no discharge from the TSF is expected. An operational emergency spillway will be available at all times during TSF operation, constructed in the embankment abutment in order to protect the integrity of the constructed embankments in the event of emergency overflow.

The closure spillway will be located at the final supernatant pond location, and will be excavated through the northern ridge line to ensure all rainfall runoff from the TSF will safely discharge after operation ceases.

Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals from the embankments and the eastern and western perimeter of the TSF.

A pipeline containment trench will be constructed during Stage 1 to contain both the tailings delivery pipeline and decant return pipeline between the TSF and Plant Site, as the flow in both pipelines is contaminated and cannot be discharged to the environment if the pipeline bursts. The pipeline trench will be situated adjacent to the main access road and will drain to designated catch ponds or the TSF.

18.4.4 Monitoring

A monitoring programme for the TSF will be developed to monitor for any potential problems which may arise during operations. The monitoring will include:

- Monitoring bores and surface water sampling stations downstream of the TSF.
- Standpipe piezometers in the embankment to monitor the phreatic surface.
- Settlement pins to check embankment movement.

18.4.5 Rehabilitation

At the end of the TSF operation, the downstream faces of the embankment (external to the Eastern Waste Dump) will have a slope of 3H:1V, with 5 m wide benches located at 10 m height intervals, for an overall slope profile of 3.5H:1V. The designed profile will be stable under seismic loading conditions, will provide a stable drainage system, and will allow for revegetation.

The TSF closure spillway will be excavated after the remaining supernatant water is proven to be suitable for release and during rehabilitation of the tailings surface subsequent to decommissioning. The closure spillway will be excavated from the final supernatant pond location, running north through the ridge-line and discharging into the adjacent (northern) catchment. Rehabilitation of the tailings surface will commence upon termination of deposition into the TSF. The final profile of the tailings surface will slope towards the closure spillway and the low spot on the tailings surface will be adjacent to the TSF northern perimeter, so that no re-shaping of the tailings surface will be required for full water shedding.

The final soil cover for the tailings surface subsequent to decommissioning will be confirmed during operation based on ongoing operational tailings geochemistry test results. The following covering for the tailings beach has been allowed for:

- Low permeability fill layer (300 mm).
- Topsoil growth medium layer (200 mm).
- The finished surface will be shallow ripped and seeded with shrubs and grasses.

18.4.6 Tailings Geochemistry

Geochemical testing was completed on tailings solids and supernatant liquor.

The results of the testing are summarised below:

- The tailings sample was classified as Acid Consuming due to a strongly negative net acid producing potential (NAPP) and net acid generating (NAG) pH of 8.3. Therefore there would be no specific controls required in relation to the acid formation potential of the tailings.

- The sulphide concentration of the tailings sample was calculated at 1.28% which is considered to be high. The maximum potential acidity was calculated from the sulphide concentration at 39 kg H2SO4 / tonne of tailings, which is also high.

- The acid neutralising capacity calculated as being available from carbonate minerals was extremely high.

- The net acid producing potential of the sample is strongly negative with a high ANC / MPA ratio, indicating that the sample has significant excess neutralising capacity.

- The results of the net acid generation test indicate that under extreme oxidising conditions, the tailings did not produce any measurable acid, and that weakly alkaline pH conditions are likely to prevail within the tailings pore waters. The acid base accounting results recorded a negative net acid producing potential of -207 kg H_2SO_4/t, with a final NAG pH of 8.3.

- The results of the multi-element analysis on the tailings solids were compared to various soil intervention guidelines. The tailings solids recorded a moderate number of element enrichments, with boron, molybdenum and sulphur all found to be highly enriched and arsenic, antimony and selenium found to be significantly enriched. Further, chromium, manganese, nickel, sulphur, sulphate and vanadium exceed the soil guideline values.

- The supernatant was found to exceed the guidelines for release / livestock water for the following parameters: arsenic, copper, cyanide (total, WAD and free), iron, molybdenum and nickel. Of these determinants, cyanide (total, WAD and free) and copper were found to exceed the guideline values by the greatest margin.

- Based on the data reviewed to date, it is considered that the proposed liner system is appropriate for the Fekola TSF.

Cyanide concentration reduces as it moves through the leaching circuit. The cyanide concentrations recorded during testwork (in the supernatant water) were 59 mg/L (CN_{total}) and 30 mg/L (CN_{wad}).

When the tailings reach the tailings storage facility, the tailings will be deposited at the perimeter of the TSF and the supernatant will run down the tailings beach to the supernatant pond. Significant decomposition of cyanide will occur through volatilisation and other processes as the supernatant runs down the beach. On reaching the supernatant pond, decay of the cyanide in the pond will continue to occur mainly as a result of the decrease in the pH of the water and the ultraviolet light falling on the pond. Without cyanide destruction measures in place, KP experience in West Africa is that supernatant ponds are typically in the range on 1 to 10 mg/L CN_{wad}. With cyanide destruction measures in place, the supernatant pond may be less than 0.5 mg/L CN_{wad}. These CN_{wad} concentrations are significantly less than the guideline value for protecting bird life (50 mg/L). Further tailings characterization is ongoing and results will be incorporated into the project as appropriate

Spillway discharge from the TSF may only occur if a storm event or rainfall sequence greater than the design criteria occurs. Therefore, allowing for cyanide decay processes and dilution by such an extreme event, the CN_{total} concentrations in any spillway flows would likely be below 0.1 mg/L.

18.4.7 Geotechnical

The geotechnical site investigation for the TSF conducted by KP included three boreholes along the TSF embankment alignment and 26 test pits in the TSF basin area.

The boreholes and test pits identified variable ground conditions, which typically comprised:

- A shallow thickness of topsoil and colluvium to a typical depth of 0.3 m and comprising very loose or loose sandy and silty gravel. Alluvium is present close to stream courses and is typically cohesive.
- A zone of lateritic residual soil comprising sandy silty gravel extending to up to 6 m depth and averaging 3 to 3.5 m, with highly varied levels of cementation. In some of the pits the material was sufficiently indurated such that it was a low and very low strength rock (Ferricrete) whilst in others the material had little cementation and retained soil-like properties.
- The ferricrete is most prominent along the line of the main embankment. It was often highly permeable and the TSF cut-off trench will need to extend through this material.
- A zone of residual soil comprising medium to high plasticity sandy silt with gravel extending from the base of the laterite material down to bedrock at typically 15 to 20 m depth.
- Bedrock was encountered in all three boreholes at a depth of between 10 m and 26 m. The encountered rock is metamorphic and either sericite or chlorite schist. The first few metres of the rock that were penetrated were typically identified to be extremely to highly weathered and very low to low strength.

Both Zone A and Zone C material are considered to be available within the TSF basin. It is considered that the cohesive colluvium, laterite gravel and residual silt material, when remoulded, should provide sufficiently low permeability to be used as Zone A material. These material types, as well as the granular colluvial material and ferricrete, are considered suitable for use as Zone C.

18.4.8 Seepage Assessment

Seepage modelling was undertaken to evaluate the phreatic conditions in the TSF embankment.

Based on the assessment, the following conclusions are made:

- The upper bound seepage through the sub-grade for the Stage 1 TSF configuration will be approximately 0.1 to 0.4 L/s.

- The upper bound seepage through the sub-grade for the final TSF configuration will be approximately 0.2 to 2.5 L/s.

18.4.9 Stability Analysis

The stability of the proposed Stage 1 and Final TSF embankment sections were assessed under static and seismic loading conditions using limit equilibrium methods. The stability of the TSF embankment was assessed in order to confirm the factors of safety against shear failure considering long-term drained conditions, short-term undrained conditions and post seismic conditions. The effect of pore water pressures on embankment stability was modelled during the stability assessment.

ANCOLD guidelines were adopted as the acceptable factors of safety for the TSF. Table 18.4.2 lists the recommended factors of safety under loading conditions. The results of stability analysis are presented in Table 18.4.3.

Table 18.4.2 Desired Minimum Factors of Safety

Loading Condition	Minimum FOS
Long-term drained	1.5
Short-term undrained (potential loss of containment)	1.5
Short-term undrained (no potential loss of containment)	1.3
Post-seismic	1.0 – 1.2

Table 18.4.3 Summary of TSF Embankment Stability Analysis Results

Location	Loading Condition	Factor of Safety
Stage 1	Long-term (drained) – circular failure	1.6
	Long-term (drained) – block failure	1.5
	Short-term (undrained) – circular failure	1.5
	Short-term (undrained) – block failure	1.5
	Post-seismic – circular failure	1.2
	Post-seismic – block failure	1.4
	Long-term (drained) – circular failure	1.9
	Long-term (drained) – block failure	2.1
	Short-term (undrained) – circular failure	1.6
	Short-term (undrained) – block failure	1.6
	Post-seismic – circular failure	1.2
	Post-seismic – block failure	1.5

Based on the stability assessment, the facility will achieve factors of safety recommended by ANCOLD and therefore should be considered as stable.

18.4.10 Decant from Tailings Storage Facility

Supernatant water will be removed from the TSF via submersible pumps located within decant towers, constructed at start-up and raised during operation.

Decants will operate automatically, reclaiming water from the TSF and pumping it via a HDPE pipeline to the process plant. The decant pumps will be raised on a regular basis to ensure that tailings do not enter the pump intake and that the good quality, recycle water is returned to the process plant.

TSF recycle rates for each year of operation are provided in Table 18.4.4. The maximum and minimum rates show the range of monthly values throughout each year. The recycle rates are expressed as a percentage of water in slurry reporting to the TSF.

Table 18.4.4 Average conditions – TSF recycle rates

Year	Average Monthly Recycle Rate (%)	Maximum Monthly Recycle Rate (%)	Minimum Monthly Recycle Rate (%)
1	49.4	66.6	42.1
2	50.6	73.7	41.2
3	51.2	73.8	40.7
4	51.7	73.8	40.3
5	52.1	73.8	40.1
6	52.4	73.8	39.9
7	52.7	73.8	39.7
8	53.0	73.8	39.6
9	53.4	73.8	39.4
10	53.7	73.8	39.2
11	54.1	73.8	39.0
12	54.4	73.8	38.9
13	40.8	41.4	39.6

18.5 Surface Water Management

18.5.1 Design Objectives

The Fekola pit footprint is located in an existing natural drainage course, with an upstream catchment of 9 km^2 requiring diversion around the pit. The site surface water management system is designed to prevent runoff from events up to and inclusive of a 1 in 100 year recurrence interval storm event from entering the pit.

The design objectives for the surface water management system are as follows:

- Diversion of rainfall runoff from upstream catchments, around site infrastructure to discharge off site downstream of the project.
- Containment of sediment-laden runoff from the site infrastructure, via collection channels, retain coarse to fine sand particles within sediment basins on site and controlled discharge from site.

18.5.2 Diversion Structures

The proposed site surface water management system will be constructed over two phases, as follows:

- Phase 1A (pre-mining): the main drainage course running through the site (between the Plant Site and TSF) will be diverted initially to the north, around the starter pit.
- Phase 1B (first 0 to 3 months after commencement of pit dewatering): the diversion channel running from the fresh water storage pond spillway, running between the plant site and the waste dump and to the south of the pit, will be required to divert WSP spillway overflow off-site.
- Phase 2 (during operation): as the pit expands to the north, the Phase 1A diversion channel will require decommissioning. Flows reporting to this diversion will be diverted to the south by raising the existing diversion bund and excavating a channel running due south to tie into the existing Phase 1B channel.

18.5.3 Collection and Control Structures

Fresh Water Storage Pond

The fresh water storage pond is the main collection and storage pond for fresh water on site, and is able to store up to 116,700 m^3 of water at the maximum operating level. The fresh water storage pond will collect rainfall runoff from the upstream catchment and surplus pit groundwater dewatering.

The fresh water storage pond embankment will be shared with the contact water storage pond, and be constructed primarily of low permeability fill material.

Water stored in the fresh water storage pond will be pumped back to the plant to supply plant fresh water requirements and process make-up water requirements, if required.

The fresh water storage pond general arrangement is shown in Figure 18.5.1 and typical embankment sections are shown in Figure 18.5.2.

Figure 18.5.1 Fresh and Contact Water Storage Ponds - General Arrangement

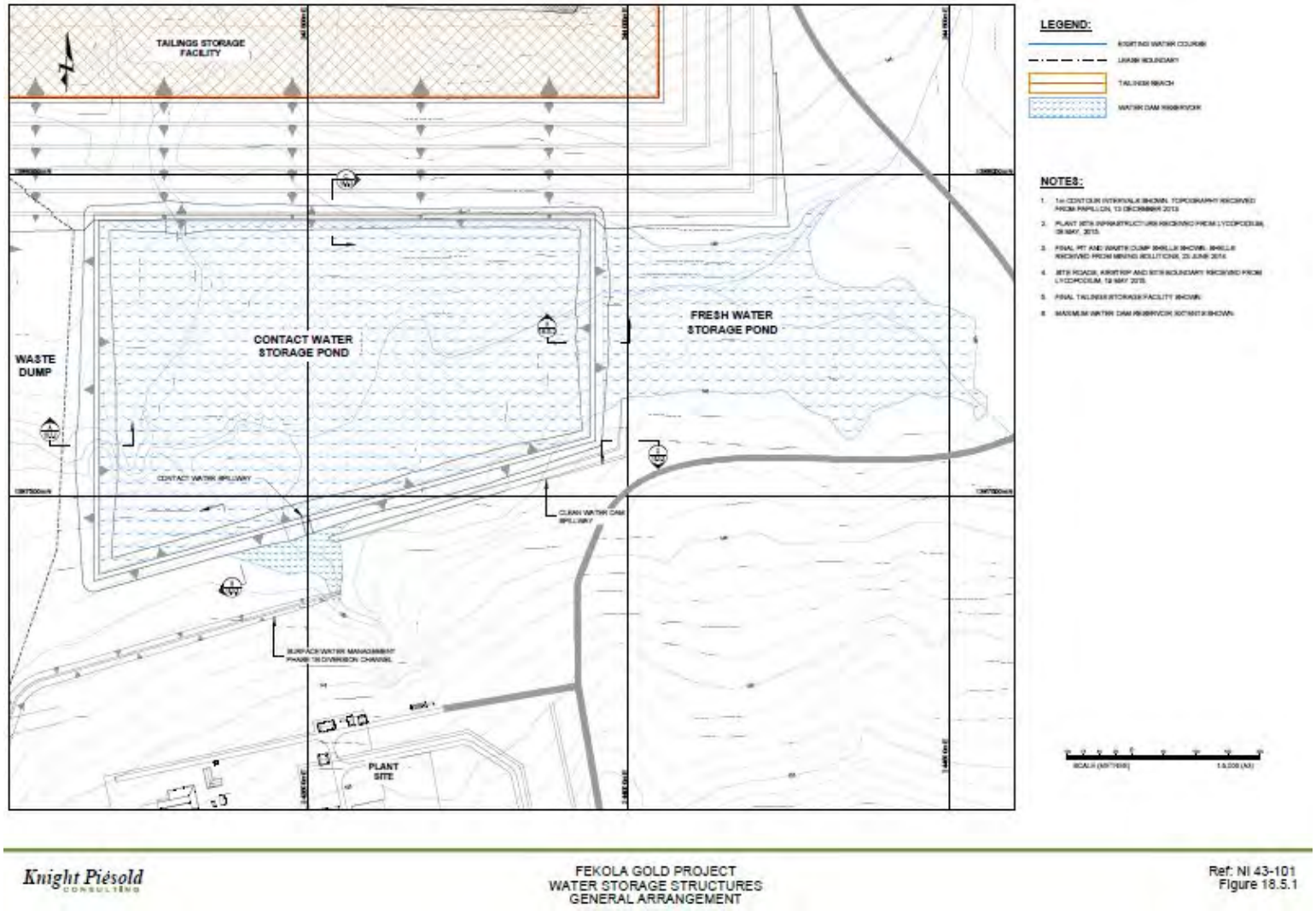
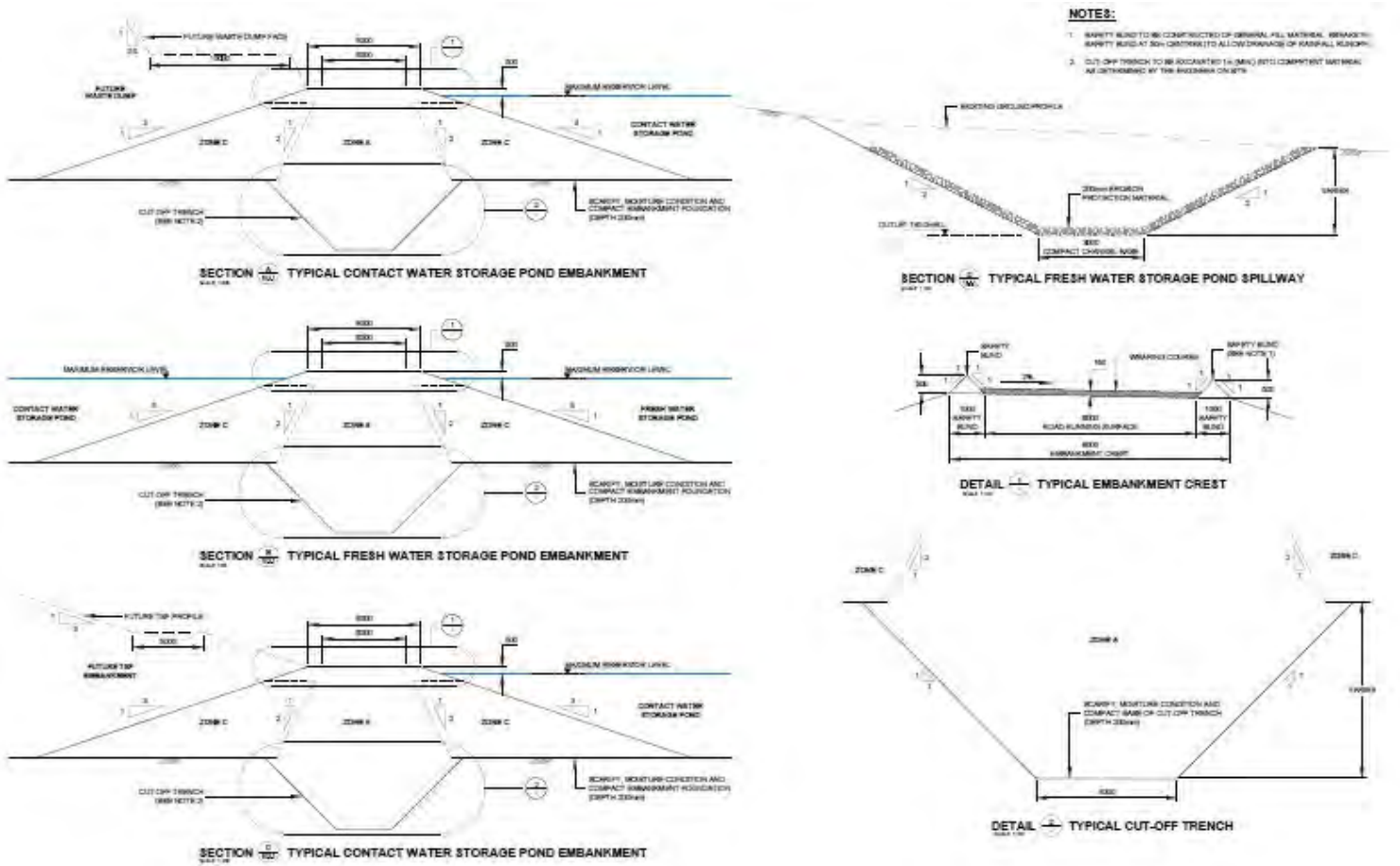


Figure 18.5.2 Fresh and Contact Water Storage Ponds - Typical Embankment Sections



18.5.4 Contact Water Storage Pond

The contact water storage pond is the main collection and storage pond for potentially contaminated water on site, and is able to store up to 2,024,000 m^3 of water at the maximum operating level.

The contact water storage pond embankment will be constructed primarily of low permeability fill material, and the contact water storage pond reservoir will be fully enclosed on all four sides.

The contact water storage pond general arrangement is shown in Figure 18.5.1 and typical embankment sections are shown in Figure 18.5.2 above.

18.5.5 Sediment Control Structures

Sediment control structures (SCSs) will be constructed in the downstream reaches of catchments impacted by site infrastructure. Discharge from the SCSs will be to the environment downstream of the project site. Water quality in the SCS reservoirs should be monitored, and the water may be pumped back to the plant for treatment (if required) or may be used for process make-up water requirements and dust suppression (if suitable).

SCSs will reduce flow velocities facilitating sediment settling. The SCSs were designed to capture particles of fine sand and coarser.

The SCS locations were selected based on the natural topography, to reduce embankment fill volumes and increase the storage capacity of each structure.

The SCSs were designed to limit maximum water depth to 1.5 m for safety reasons.

18.5.6 Falémé River Flood Assessment

In order to assess flooding of the Falémé River, particularly with concern to the potential for flooding of the proposed Fekola pit, flood modelling was completed by KP.

The flood modelling incorporated the following information:

- A hydrologic rainfall / runoff model to determine the additional flows contributing to the peak flow of the Falémé River downstream of the Fadougou river level gauge.
- A hydraulic water surface profile model to generate an estimated rating curve, and determine the depth and areal extent of inundation resulting from the 1% Average Exceedance Probability (AEP) river flow plus the 1% AEP storm.

Results from the inundation model were used to determine whether any specific infrastructure, such as a flood protection bund, would be required to protect the pit.

The following input data were used in the modelling:

- KP was provided with daily average flow levels for the World Meteorological Organisation (WMO) operated Fadougou river gauge for the time period of 1952 through to 2011 (total 60 years). However due to missing data, only 41 years of data were considered valid for use in the flood frequency analysis. KP was unable to obtain a rating curve for the station.
- Historical precipitation data were sourced from the Kéniéba climate station (refer Section 2 for further information).
- The synthetic topographic model used for the flood modelling was a combination of the following:
  - LIDAR survey data for the Project area
  - publicly available Shuttle Radar Topography Mission (SRTM) where the LIDAR data were not available (significant portions of the river bed)
  - a single cross section of the river at the Fadougou gauge location
  - a long section of the river bed in proximity to the Fadougou gauge
  - additional survey information for the river banks for the Fadougou gauge.

In order to estimate the 1 in 100 year ARI flood extents for the Falémé River, both hydrologic and hydraulic models were utilised:

- The hydrological conditions of the Falémé River comprise the flow rate estimated at the Fadougou gauge station, and the additional flows from the small catchments downstream of the gauge.
- In order to estimate the hydraulic behaviour of the Falémé River adjacent to the Fekola site, a hydraulic water surface profile model was constructed.

As a rating curve for the Fadougou river gauge station was not available, the Falémé River cross-section survey was used to generate a rating curve, to allow depth measurements to be converted into flow rates.

The predicted flood inundation level for the section of the Falémé river adjacent to the project (in particular at the Pit), does not encroach on the current pit outline. However, the sediment control structure immediately downstream of the waste dump (located to the south west of the Pit) may require erosion protection on its downstream face.

Once the waste dump is in place, a peak flood level of approximately 130.5 m would be required to result in flooding. Based on the frequency analysis of the flood levels, a flood resulting in a peak flood level of 130.5 m is estimated to be greater than a 1 in 1,000 year recurrence interval.

18.6 Water Supply

Water supply for the Project is grouped into the following sources:

- Pit Groundwater.
- Surface Water.
- Falémé River Bores.
- Potable Water Bores (Covered in Section 18.9).

18.6.1 Pit Groundwater

Knight Piésold (KP) undertook a groundwater investigation of the proposed open pit at Fekola. The field investigation results together with monitoring data and field investigation data previously collected by Papillon were used to develop a numerical groundwater flow model used to predict groundwater inflows to the various stages of the Fekola pit.

The predicted pit inflows are summarised as follows:

- Average climatic conditions (base case) – the average groundwater inflow rate over the 11 year period is predicted to be 129 L/s.
- Groundwater inflows are predicted to reach 127 L/s by the end of Year 1 and increase to 151 L/s by Year 4 and then peak to 170L/s by the end of Year 9.

An assessment of baseline groundwater quality data for the Project was completed by KP.

The sample locations include groundwater samples from village wells. In April 2013, groundwater samples were also taken from regional test bores installed by Papillon some of which are in the mine area.

The baseline water quality collected to date indicates that the groundwater is for the most part pH neutral, fresh and of good quality. The main exceptions are as follows:

- Arsenic occurs at low concentration levels but above the WHO (2011) drinking-water guideline value of 0.01 mg/L drinking-water guideline.
- Nitrate concentrations are consistently above WHO (2011) drinking water guidelines in the Tintiba village well (probably due to poor sanitation).

Pit ground water will be extracted by pit dewatering bores located around the perimeter of the open pit extent. Water will be pumped either direct to the process plant fresh water tank or to the fresh water storage pond for storage.

18.6.2 Surface Water

Rainfall runoff from the upstream catchment will be collected and stored in the fresh water storage pond, which is sized to store up to 116,700 m^3 of water at the maximum operating level.

Water stored in the fresh water storage pond will be pumped to the process plant fresh water tank to supply plant raw water requirements and process make-up water requirements, if required.

An assessment of baseline surface water quality data for the Project was completed by KP.

The baseline water quality collected to date indicates that the surface water is for the most part pH neutral, fresh and of good quality. The main exceptions are as follows:

- Arsenic occurs at low concentration levels but above the WHO (2011) drinking-water guideline value of 0.01 mg/L drinking-water guideline.
- Iron, cadmium and lead also occur at low concentrations but above their respective WHO (2011) drinking-water guideline values in several surface water samples.

18.6.3 Falémé River Bores

Three additional bores have been included at the Falémé River in the event that site water quantity or quality requirements are not met as anticipated by the Pit Dewatering Bores. Location of these bores and extraction rates will be determined if they are required in the future.

18.6.4 Water Balance

Based on the water balance modelling, a number of conclusions can be drawn as follows:

- The TSF has sufficient capacity to store all design rainfall events.
- Make-up water is required from the fresh water storage pond during each year of operation for average and design dry rainfall sequences. In these instances, the model supplies all required make-up water from the fresh water storage pond (supplemented by pit dewatering). The magnitude and duration of shortfall increases with drier conditions.
- Modelling results indicate that no plant water shortfalls occur during operation for all average and design dry conditions due to the availability of the predicted pit dewatering. As such, no additional water supply is required. This is primarily due to the volumes of water pumped to the fresh water storage pond from the pit dewatering. Should this source of water become unavailable, or is significantly lower than anticipated, the rainfall runoff from the surrounding catchments of the fresh water storage pond would not be sufficient to supply the process plant water requirements during average or drier conditions. Alternative water sources (e.g. alluvial borefield at the Falémé River) would need to be located if pit dewatering was not available, (or not be of suitable quality).

- The maximum annual spillway discharge rate for the contact water storage pond, and therefore the maximum potential treatment volume is 454,000 m^3/year for average conditions and 1,149,000 m^3/year for wet conditions. The maximum monthly spillway discharge rate for the contact water storage pond, and therefore the maximum potential treatment volume is 316,000 m^3/month for average conditions and 521,000 m^3/month for wet conditions.

18.7 Power

Due to the remote location of the Fekola site and questionable guarantee of supply, connection to the Malian grid is not a feasible option. Power will therefore be generated by a dedicated power station purpose built for the project.

18.7.1 Power Generation

Power supply to the site will be from a combination HFO and diesel fuelled power station located adjacent to the process plant. The power station will supply the main HV switchroom inside the processing plant from which power will be distributed.

The power station has been sized to accommodate a maximum demand power draw of 29.4MW with a configuration as follows:

- 7 x 6 MW 13.8kV medium speed generators (5 duty, 2 standby).
- 5 x 1.6 MW 13.8kV high speed generators (emergency standby).
- 13.8 kV switchroom.
- Black start generator.
- The SAG and Ball mills at the process plant are by far the largest loads on the power station. The SAG Mill has been specified with a VVVF (variable speed) drive and the Ball Mill an LRS to provide a 'soft start' capability to reduce the load surge during start-up and minimise the need for 'spinning reserve' in the power station.

18.7.2 Electrical Distribution

The electrical system is based on 13.8 kV distribution and 480 V, 60 Hz working voltage.

The 13.8 kV feeder from the power station will feed the site distribution 13.8 kV switchboard.

For the process plant the 13.8 kV supply will be stepped down from 13.8 kV to 480 V at each switchroom using separate 13.8 kV / 480 V distribution transformers fed from the HV distribution board.

Four switchrooms are located throughout the process plant, nominally covering:

- Primary Crushing.

- Milling.
- Leach / Thickening / Water and Air Services.
- CIP / Elution / Goldroom.

Switchrooms will house 480 V motor control centres (MCCs), area VVVF drives, plant control system cabinets, plant lighting transformers, various distribution boards and UPS power distribution.

Power distribution to plant and mine infrastructure will be via kiosk substations fed from the HV distribution board. Kiosk substations will be provided for:

- Laboratory.
- Process Plant Workshop / Warehouse.
- Process Plant and Admin Offices / Training Buildings.
- Mine Services Area.

13.8 kV overhead power lines will provide power to various remote facilities (TSF pumps, bore pumps, water storage pond pumps, etc.). Pole mounted transformers will step down the voltage at each location and supply an outdoor 480 V switchboard local to each equipment area.

Accommodation camp power will be supplied from a local pad mounted transformer fed from the 13.8 kV overhead line.

18.7.3 Electrical Buildings

Electrical buildings will pre-fabricated ‘flat pack’ panel buildings to minimise installation time on site. Buildings will be installed on a structural framework over 2 m above ground level to allow for bottom entry of cables into electrical cabinets.

The electrical buildings will be installed with air-conditioners and suitably sealed to prevent ingress of dust.

18.7.4 Transformers and Compounds

All the 13.8 kV / 480 V distribution transformers will be of ONAN cooling configuration and vector group Dyn11.

Fire rated concrete walls will be constructed around the pad mounted transformers.

18.8 Fuel Supply

The powerstation includes tankage for fuel storage capacity of 1,100 kL of diesel and 4,000 kL of HFO. This equates to 30 days of fuel storage for the powerplant operating under full load conditions.

An additional two 1,200 m^3 diesel storage tanks are provided for fuel usage for the mobile fleet. At an estimated operating fuel consumption of 1,200 kL/month, this equates to 30 days of site diesel storage at full production.

18.9 Potable Water

The potable water demand for the Project has been calculated on a per capita usage basis and is summarised in Table 18.9.1 below.

Table 18.9.1 Potable Water Demand

Area	No. of Personnel	Usage (L/person/day)	Demand (m^3/day)
Accommodation Camp	411	300	123.3
Plant Drinking & Ablutions	500	75	37.5
Laboratory & Safety Showers	-	-	31.2
Total Process Plant			**68.7**

Separate potable water systems will be provided for the accommodation camp and the process plant usage.

18.9.2 Process Plant

Fresh water will be sourced from bores located around the site and pumped to a potable water feed tank located adjacent to the potable water treatment facility at the process plant.

A vendor package modular potable water treatment plant including filtration, reverse osmosis (RO) and ultra-violet sterilisation will be installed. Potable water will be stored in the plant potable water tank and will be reticulated to the plant buildings, site ablutions, safety showers, and other potable water outlets.

18.9.3 Accommodation Camp

Fresh water will be sourced from bores located around the site and pumped to a potable water feed tank located adjacent to the camp potable water treatment facility at the accommodation camp.

A vendor package modular potable water treatment plant including filtration, ultra-violet sterilisation and chlorination will be installed. Potable water will be stored in the camp potable water tank which will provide two day reserve storage in the event of supply line disruptions. Water will be delivered into the camp reticulation system using a constant pressure variable flow pump system. The pump skid will include a UV disinfection unit to provide additional security against contamination.

18.10 Sewage and Solid Waste Management

18.10.1 Sewage Treatment

Effluent from all water fixtures in the process plant, mine services area and administration areas will drain to gravity sewerage systems. The gravity sewerage system for each area will drain to a sewer pump station from where it will discharge via a pressure main to a vendor package sewage treatment plant system located adjacent to the process plant.

Treated effluent will be discharged to an evaporation pond. Treatment plant sludge will be suitable for direct landfill burial.

18.10.2 Solid Wastes

Wastes will be sorted and reused or recycled as far as the limited access to recycling facilities allows.

Waste lubricating oils will be returned to the supplier or appropriate for recycling.

General non-hazardous solid wastes will be deposited into a suitable mine waste dump and promptly covered to deter vermin and scavengers.

Dangerous or hazardous waste will be collected and stored briefly before being transferred to a suitable permitted facility, either on-site or off-site depending on the specific materials and requirements.

18.11 Accommodation Camp

The accommodation camp will be located to the south of the process plant and will provide accommodation for International Expatriate and African national staff not originating from the local area.

The camp will consist of the following:

- Accommodation for 411 persons, including:
  - one VIP accommodation dormitory, consisting of 8 x single bed rooms with ensuite and lounge area
  - two Executive accommodation dormitories, each consisting of 24 x single bed rooms with ensuite
  - seven Senior accommodation dormitories, each consisting of 31 x single bed rooms with shared ablution facilities
  - three Double accommodation dormitories, each consisting of 23 x 2 bed rooms with shared ablution facilities.

- Dry mess / kitchen with storage and preparation, kitchen and dining facilities for 324 people.
- Wet mess.
- Medical facility.
- Administration Office / Store building.
- Mud / Locker entry room.
- Fitness centre.
- Indoor Gymnasium, with Basketball court and running track.
- All buildings are connected by a common hallway with multiple, TV, games and reading rooms included as part of the main traffic thoroughfare.

18.12 Plant Facilities

18.12.1 General

Site buildings will be 'fit for purpose' industrial type structures.

Workshops, warehouses and reagent storage sheds will be constructed of a concrete slab on ground with structural steel frame and metal cladding.

Offices and amenity buildings will be prefabricated structures for ease of site installation and fit-out.

18.12.2 Mine Services Area Facilities

The Mine Services Area will be located to the North West of the process plant for ease of access to the pit and mine work areas. Building infrastructure and facilities within the mine services area will include:

- Security gatehouse with entry boom gate control.
- Mining administration office / training room and ablutions.
- Mine changehouse.
- Mine workshop / warehouse.
- Heavy vehicle washbay facility.
- Bulk fuel storage and dispensing facilities.

18.12.3 Process Plant Area

The following building infrastructure will be located inside the process plant high security area:

- High security gatehouse.
- Camp access gatehouse.
- Process plant offices.
- Plant training room.
- Plant changehouse.
- Plant ablutions.
- Lime reagent storage shed.
- Cyanide, caustic, lead nitrate and flocculant storage shed.
- SMBS, copper sulphate and acid storage shed.
- Plant laboratory.
- Plant workshop / warehouse.

18.12.4 Other Support Facilities

The following building infrastructure will be located inside the low security area on entrance to the Fekola site to the north of the process plant:

- Low security gatehouse.
- Administration office / training room and ablutions.
- Medical / First Aid Clinic.

FEKOLA GOLD PROJECT

NI 43-101 -MARKET STUDIES AND CONTRACTS

3174-STY-002

Table of Contents

			Page
19.0	**MARKET STUDIES AND CONTRACTS**		**19.1**
	19.1	Market Studies	19.1
	19.2	Contracts	19.1

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No formal market studies have been undertaken.

A gold price of US$ 1,300/oz based on analyst consensus has been used for the Mineral Reserve estimate and the economic analysis.

Fekola will produce gold doré which is readily marketable on an 'ex-works' or 'delivered' basis to a number of refineries in Europe and Africa. There are no indications of the presence of penalty elements that may impact the price or render the product unsalable.

Payment terms are widely available in the public domain and vary little from refinery to refinery. B2Gold has contracts in place for sale of gold from its four producing mines in Nicaragua, the Philippines and Namibia. It has been assumed that Fekola doré will attract similar terms.

19.2 Contracts

There are no material contracts or agreements in place as of the effective date of this report.

Refining contracts are typically put in place with well recognized international refineries and sales are made based on spot gold prices. These contracts typically include fees for transportation of the product from the site, insurance, assaying, refining and an allowance for metal losses during refining. The ability to get a contract in place for the sale of doré prior to start of production is not seen as a risk to the Project.

FEKOLA GOLD PROJECT

NI 43-101 - REQUIRED PERMITS AND ENVIRONMENTAL CONSIDERATIONS

3174-STY-002

Table of Contents

			Page
20.0	**ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT**		**20.1**
	20.1	Environmental Studies	20.1
	20.2	Permitting	20.1
		20.2.1 Environmental Permit	20.1
		20.2.2 Environmental Permit for Access Road	20.2
		20.2.3 Mining Permit	20.2
		20.2.4 Approval of Community Development Plan	20.2
		20.2.5 Approval of Mine Closure Plan	20.2
		20.2.6 Additional Permits and Authorizations	20.3
	20.3	Environmental Impact Assessment Process	20.4
	20.4	Summary of Environmental and Social Setting	20.5
		20.4.1 Physical Setting	20.5
		20.4.2 Biological Setting	20.9
		20.4.3 Social Setting	20.10
	20.5	Summary of Key Environmental and Social Potential Impacts	20.13
		20.5.1 Air quality	20.14
		20.5.2 Hydrology, Hydrogeology and Water Quality	20.15
		20.5.3 Biodiversity	20.17
		20.5.4 Access to Land Resources	20.19
		20.5.5 Livelihood and Employment	20.21
		20.5.6 Social Services / Infrastructure	20.22
	20.6	Summary of Environmental and Social Management and Monitoring Plan	20.23
		20.6.1 Environmental Monitoring Plan	20.24
		20.6.2 Water Management Plan	20.24
		20.6.3 Waste Management Plan	20.24
		20.6.4 Erosion and Sediment Control Plan	20.25
		20.6.5 Rehabilitation and Preliminary Mine Closure Plan	20.25
		20.6.6 Emergency Preparedness and Response Plan	20.25
		20.6.7 Biodiversity Action Plan	20.25
		20.6.8 Stakeholder Engagement Plan	20.25
		20.6.9 Community Development Plan	20.26
	20.7	Mine Rehabilitation and Closure	20.26

TABLES

Table 20.2.1	Summary of Additional Permits and Authorizations Required for the Project	20.3
Table 20.3.1	Specialist Studies Commissioned as part of the 2013 ESIS	20.4
Table 20.3.2	Specialist Studies Commissioned as part of the ESIA Update	20.5
Table 20.7.1	Summary of Estimated Rehabilitation and Closure Costs	20.27

FIGURES

Figure 20.4.1	Local Drainage Map of the Fekola Gold Project Area and Surrounds	20.8
Figure 20.4.2	West Sudanian Savannah Ecoregion (Yellow)	20.9
Figure 20.4.3	Proximity Map of Villages in the Area Surrounding the Project	20.12

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

An Environmental and Social Impact Assessment (ESIA) was originally completed for the Project in 2013 (2013 Environmental and Social Impact Statement (2013 ESIS)). This 2013 ESIS was approved by the Ministry of Environment and Sanitation on 29 April 2013. B2Gold has commissioned an update of the ESIA (ESIA Update) to fill gaps identified in the 2013 ESIS, reflect optimization improvement and modifications to the Project design, assess these improvements and modifications for their potential impacts against baseline conditions in the Project area, and align the assessment with international standards including the IFC environmental and social performance standards. The ESIA Update is scheduled to be completed on 30 June 2015.

The ESIA Update contains the policy, legal and administrative framework under which the study was carried out and the Project will be regulated and managed. It includes a description of the Project in its geographic, ecological, social and temporal context. The ESIA Update includes baseline data describing relevant physical, biological and social conditions associated with the Project area and identifies the likely types of environmental and social impacts associated with the construction, operation and closure of the Project. The ESIA Update assesses the magnitude and likelihood of these impacts based on currently available Project information and presents proposed mitigation measures needed to minimize potential impacts to acceptable levels. Stand-alone management plans to address residual impacts are also provided as part of the ESIA documentation.

20.2 Permitting

Various permits and authorizations are required for the Project. The following sub-sections provide a brief summary of the status of key permits.

20.2.1 Environmental Permit

The environmental permit for the Project was granted to Songhoi Resources SARL by the Ministry of Environment and Sanitation via Decision No. 2013-0033MEA-SG on 29 April 2013. The permit requires that Songhoi begin construction of the Project within 3 years of the issue of the permit and also allows the government to perform an environmental audit of the Project every five years. In addition, the permit includes six clauses with conditions/requirements relating to the following:

- Air quality.
- Soil conservation.
- Surface water and groundwater quality.
- Noise and safety.
- Cultural heritage.

- Land appropriation.

It should be noted that the environmental permit was granted based on the previous ESIS dated 29 April 2013. This ESIA Update report incorporates the content of the 2013 ESIS while also including assessment of modifications and improvements made to the Project design since the issue of the 2013 ESIS.

20.2.2 Environmental Permit for Access Road

A second environmental permit, for the access road from the Millennial Highway near Segondo village to the mine site, was granted to Songhoi by the Ministry of Environment and Sanitation via Decision No. 2015-0002MEADD-SG on 16 March 2015. Similar to the environmental permit approval for the Project, the access road approval requires that the road be constructed within three years of the issue of the permit and allows the government to perform and environmental of the access road every five years. In addition, the access road permit requires that the requirements of the Project main environmental permit be met, as well as five clauses specific to this authorization relating to:

- Air quality.
- Soil conservation.
- Surface water and groundwater.
- Health, safety, employment and quality of life.
- Cultural heritage.

20.2.3 Mining Permit

The mining permit for the Project was granted to Songhoi Resources SARL by the Secretary General via Decree No. 2014-0070/PM-RM, dated 13 February 2014, and signed by the Minister of Mines and the Prime Minister of Mali. The permit requires that Songhoi begin construction of the Project within three years of the issue of the permit, is valid for up to 30 years and requires renewal every ten years.

20.2.4 Approval of Community Development Plan

As part of the ESIA process, the Project is required to submit for approval a Community Development Plan. A Community Development Plan, dated 24 June 2013, was submitted to the Ministry of Land Administration, Decentralization and Regional Planning and approved by the Kéniéba Prefect authorities via Decision No. 13-041/PCK, dated 04 July 2013.

20.2.5 Approval of Mine Closure Plan

As part of the process for the granting of an exploitation permit under the 2012 Mining Code, the Project is required to submit a mine closure plan. A mine Conceptual Closure Plan, dated April 2013, was submitted to the Ministry of Mines as part of the exploitation permit application. By

granting the Médinandi Mining Permit to Songhoi, the Ministry of Mines and the government of Mali approved all documents submitted in support of the exploitation permit application including the submitted Conceptual Closure Plan.

20.2.6 Additional Permits and Authorizations

Several additional permits and authorizations are required for the Project. A brief summary of these permits is presented in Table 20.2.1.

Table 20.2.1 Summary of Additional Permits and Authorizations Required for the Project

Permit / Authorization	Ministry / Department	Comment / Status
Water abstraction from Falémé River	OMVS	Not currently obtained[1]
Water abstraction via pit and borehole dewatering	Ministry of Environment and Sanitation	Not currently obtained[1]
Operations Protection Area	Ministry of Mines	Delineation of Project operational boundary, Not currently obtained[1]
Airfield construction	Ministry of Equipment and Transportation	Not currently obtained[1]
Authorization for national flights at Fekola	Ministry of Equipment and Transportation	Not currently obtained[1]
Medical Clinic operation authorization	Ministry of Health	Not currently obtained[1]
Camp construction	Ministry of Housing, Urban Development and Land Affairs	Not currently obtained[1]
Sand and gravel extraction	Ministry of Environment and Sanitation	Not currently obtained[1]
Radio license	Ministry of Communication	Not currently obtained[1]
Fuel storage permit	Ministry of Mines	Not currently obtained[1]
Solid waste management authorization	Ministry of Environment and Sanitation	Not currently obtained[1]
Landfill environmental permit	Ministry of Environment and Sanitation	Not currently obtained[1]
Wastewater discharge permit	Ministry of Environment and Sanitation	Not currently obtained[1]
Chemical use approval	Ministry of Environment and Sanitation	Not currently obtained[1]
Chemical storage authorization	Ministry of Environment and Sanitation	Not currently obtained[1]
Explosives use authorization	Ministry of Mines	Not currently obtained[1]
Authorization to clear protected tree species	Ministry of Environment and Sanitation (Regional)	Completed for No Go Zone
Easements along Falémé River	OMVS	Not currently obtained[1]
Authorization for power production	Ministry of Energy	Not currently obtained[1]

[1] **Not obtained as of May 8, 2015.**

This table is not a complete list of the permits and authorizations required for the Project.

Other than as outlined above, B2Gold is not aware of significant permits or environmental factors that may affect the right or ability to commence construction activities on site. Additionally, B2Gold is not aware of reasons why additional operational permits will not be granted. B2Gold confirms that there are currently no requirements to post reclamation bonds.

20.3 Environmental Impact Assessment Process

The 2013 ESIS process was conducted to comply with Malian legislation while also taking into consideration international guidelines such as the Equator Principles and the International Finance Corporation (IFC) performance standards. The ESIA Update was completed to fill gaps identified in the 2013 ESIS, reflect optimization improvement and modifications to the Project design, assess these improvements and modifications for their potential impacts against baseline conditions in the Project area, and align the assessment with international standards including the IFC environmental and social performance standards.

The 2013 ESIS and the ESIA Update were developed by B2Gold (2013 ESIS was completed by Papillon Resources, since merged with B2Gold) with the support of environmental and social specialists that were selected on the basis of their expertise in the development of environmental and social impact assessments of mining projects and of the Project area.

The initial baseline specialist studies were conducted in approximately 2011 and 2012. These studies were incorporated into the 2013 ESIS submitted to the Government of Mali and approved on 29 April 2013.

Specialist studies conducted for the 2013 ESIS are listed in Table 20.3.1.

Table 20.3.1 Specialist Studies Commissioned as part of the 2013 ESIS

Specialist Study Title	Author
Public Participation Provisional Report	ESDCO (Environment & Social Development Company Sàrl)
Project Study of Socio-Economic Base in The Area of Influence and Summary of Public Participation Process - Interim Report	ESDCO (Environment & Social Development Company Sàrl)
Assessment Of Geochemical Test Data – Fekola Mine	The Moss Group
Fekola Biodiversity Baseline Study & Impact Assessment, Fekola Project, Fadougou, Mali	ECOREX Consulting Ecologists CC
Noise Impact Assessment for the Proposed Fekola Gold Project in Mali	Airshed Planning Professionals (Pty) Ltd
Médinandi Mine – Numerical Flow and Transport Model	Delta H (Delta-H Water System Modelling PTY Ltd)
Blasting Impact Assessment for the proposed Fekola Gold Mine Project	Anthony John Rorke
Air Quality Impact Assessment For The Proposed Fekola Gold Mine Project in Mali	Airshed Planning Professionals (Pty) Ltd
Health Management Plan : Assessment of current health risks and recommendations	Dr Dave Barnes

Additional specialist studies listed in Table 20.3.2 were commissioned as part of the ESIA Update process to assess optimization or modifications to the Project or where gaps in data were identified.

Table 20.3.2 Specialist Studies Commissioned as part of the ESIA Update

Specialist Study Title	Author
Water Baseline and Assessment	Earth Systems Pty Ltd
Priority Habitat and Species Study	Earth Systems Pty Ltd
Fish and Aquatic Species Study	INGRID Filiale du Groupe Sapiens International
Air Quality Study Update	Airshed Planning Professionals (Pty) Ltd
Noise Study Update	Airshed Planning Professionals (Pty) Ltd
Mine Materials Geochemistry Report, Water Balance Report and Fate and Solute Transport Model Report	Knight Piésold and Co.
Meteorology Baseline Report	Earth Systems Pty Ltd
Management Plans: Environmental and Social Management Plan; Environmental Monitoring Plan; Rehabilitation and Preliminary Mine Closure Plan; Water Management Plan; Erosion and Sediment Control Management Plan; Waste Management Plan; Biodiversity Action Plan; Emergency Preparedness and Response Plan; Stakeholder Engagement Plan; and Community Development Plan.	B2Gold / ESDCO / Earth Systems

To ensure that all parties are provided with an opportunity to obtain information on the Project as well as comment and raise issues relating to the proposed development, a participatory and inclusive public consultation process was developed and undertaken for the Project as part of the development of the 2013 ESIS. The emphasis of the consultation was on the effective involvement of the natural or legal persons likely to be affected by the proposed development. The consultation process focused on the communities and settlements in the license area and along the road from the town of Kéniéba to the site.

20.4 Summary of Environmental and Social Setting

20.4.1 Physical Setting

The Project is located in the South-Western part of Mali, close to the border with Senegal to the West and Guinea to the South. It is approximately 400 km (525 km by road) west of the capital city of Bamako, within the Kéniéba gold belt region. The landscape is characterized as tropical savannah, with plateaus interspersed approximately 30 to 40 m above low-lying floodplains of the Falémé River. Elevation of the Project area ranges from approximately 130 to 190 m above sea level.

The Project region has a sub-tropical climate, with relatively high and uniform temperatures and distinct seasons; wet season (July to Sept) and the dry season (October to June). The mean total annual rainfall measured at Kéniéba Station, located 38 km north of the Project, was 1,086.2 mm over 44 years. A meteorological station within the Fekola Camp has shown a close correlation to the data from Kéniéba.

The geology of the Project area is comprised of metasedimentary rock and granitoids of the Falémé Series (Paleoproterozoic Birimian age geologic parent material). The Falémé Series, which occurs within the Dialé–Daléma sedimentary basin, is comprised of carbonate rich sedimentary rocks, minor basalts and andesites, rare rhyolites, and syntectonic granitoids (Hirdes and Davis, 2003).

Geochemical characterisation testwork indicates that the mine materials (waste rock, tailings, low grade ore and construction materials) are likely to be non-acid forming and not likely to leach significant quantities of metals/metalloids to the receiving environment (Knight Piésold, 2015).

The Project is located adjacent to the Falémé River, a major tributary to the Senegal River. The river width is variable from 10 m to 45 m in the vicinity of the Project. There is significant variation in seasonal flows of the Falémé River with approximately 90% of annual flow occurring during the wet season months. Flows at Fadougou gradually decrease over the dry season from November (median flow 35.5 m^3/s) to very little flow at the end of the dry season in the months of March (median flow of 5.6 m^3/s) and April (median flow 3.9 m^3/s). Annual flow is also highly variable year to year.

There are a number of ephemeral creeks that drain the Project area, all of which drain to the Falémé River (Figure 20.4.1). Key ephemeral creeks in the Tenement include:

- Drainage 1 - located in the northern extent of the Médinandi Tenement.
- Drainage 2 - located to the west of Fadougou Village.
- Drainage 3 - which passes through the proposed location of the mine pit, waste rock dump, low grade ore stockpile, TSF, and water supply dams.
- Drainage 4 - located to the south of the No-Go Zone.

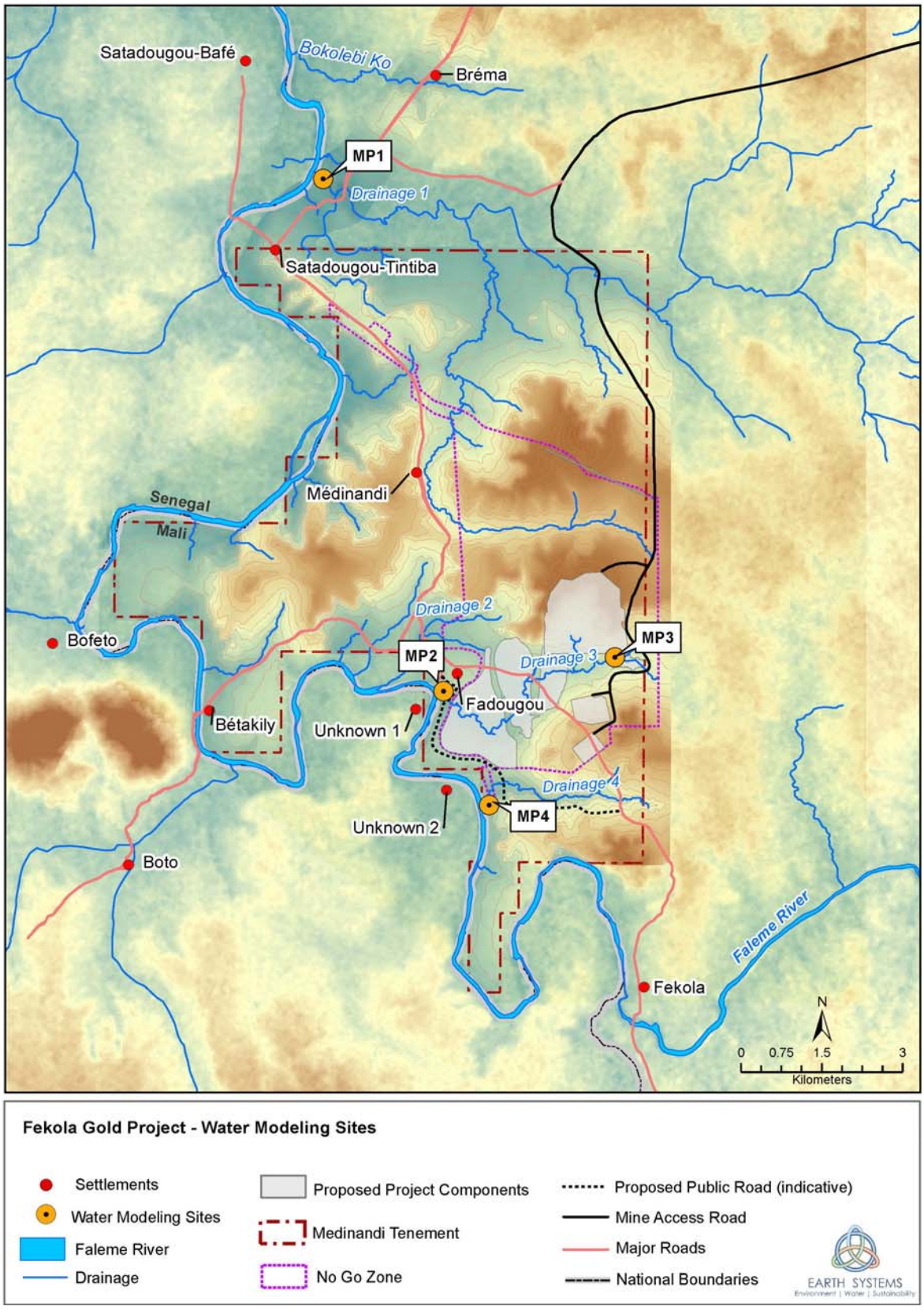
Groundwater storage in the vicinity of the Project Area is highly dependent upon rainfall and evaporation balances. Recharge of aquifers is expected to occur during the wet season when rainfall exceeds the evaporation rate, while discharge is expected to occur via evapotranspiration and the river bed. Groundwater bores around the proposed pit on the Project site showed groundwater is reached at 2 to 5 m below ground level, with observed seasonal variation of approximately 4 to 5 m. There is no significant abstraction of groundwater in the Project area; however villages in the vicinity rely on bore holes for domestic water use, drinking water and artisanal mining.

Water quality of the Falémé River was found to be good; with near neutral pH, low conductivity and low total metal concentrations. The only exception found was high total suspended solids (TSS) found locally at the monitoring site adjacent to Fadougou Village. Groundwater was also found to

be of good quality with only one exception; nitrate concentration exceeding applicable drinking water quality standards at Tintiba. Elevated nutrient concentrations in the Tintiba groundwater sample are likely to be associated with aquifer contamination from disposal of village and sewage wastes close to the village borehole.

Baseline air quality and noise in the vicinity of the Project area are typical of rural areas in the region, and no major industrial sites are located nearby. Air quality is most likely affected by sources such as: dust from unpaved roads and Harmattan winds, burning, open cooking, livestock farming, and mining. Noise emissions in the vicinity of the Project area largely originate from human activity, primarily vehicle and motorcycle use on roads and pathways and artisanal mining, both domestic and wild animals and insect activity. Baseline noise monitoring was undertaken during the wet season. The ambient (LAeq) daytime noise levels in Fadougou Village were recorded to slightly exceed the IFC ambient daytime noise criterion of 55 dB(A), while night-time ambient noise levels were slightly below IFC criteria.

Figure 20.4.1 Local Drainage Map of the Fekola Gold Project Area and Surrounds

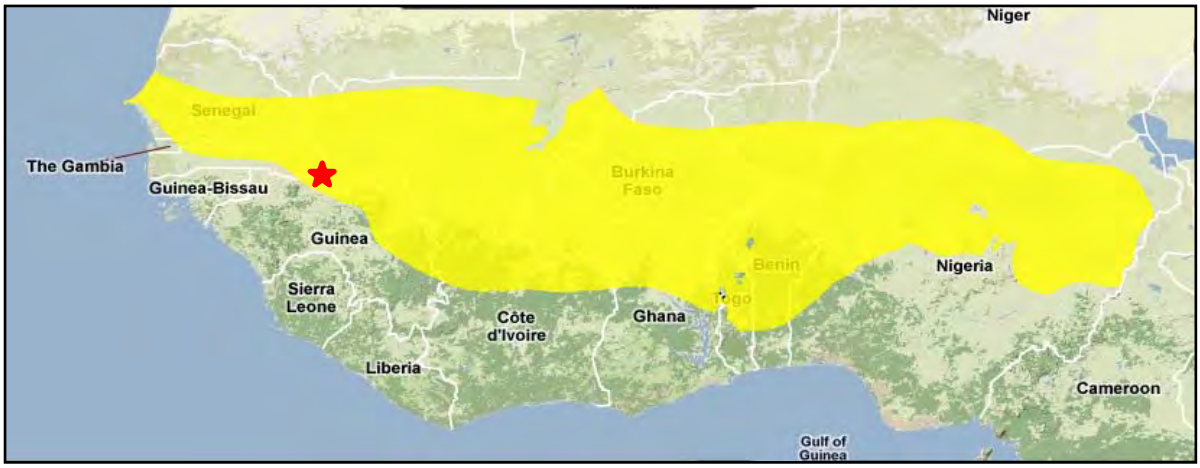


20.4.2 Biological Setting

A Biodiversity Baseline Study and Impact Assessment was conducted by Epoch Resources in January 2013 for the Project's 2013 ESIS. To supplement this information, aquatic ecology surveys in the vicinity of the Project were recently undertaken by INGRID S/C Groupe Sapiens International, and a Priority Habitats and Species Study was conducted by Earth Systems as part of the ESIA Update to provide supplementary ecological information and mapping from recent satellite imagery, current and past studies, literature, and published documents.

There are no protected areas within or near to the Project Area. The Project is located in the western part of the West Sudanian Savannah ecoregion (1,638,426 km^2). This broad ecoregion is covered by Sahelo-Sudanian / Sudanian vegetation consisting of savannah types, gallery forests, woodland and an understorey of shrubs, grasses and herbs. Vegetation is highly influenced by the two distinct seasons of the ecoregion; the hot / dry season and the wet season. The habitats in this ecoregion have been greatly reduced and fragmented by agricultural activities, fire, and timber harvesting, while populations of most of the larger mammal species have been decimated by over-hunting.

Figure 20.4.2 West Sudanian Savannah Ecoregion (Yellow)



Red star indicated approximate Project location (adapted from WWF,2015)

The aquatic environments of the Project are within the 'Senegal-Gambia freshwater ecoregion'. This freshwater ecoregion consists of tropical and subtropical floodplains, rivers and wetland complexes. The Senegal-Gambia freshwater habitats are considered nationally important, and are also considered 'vulnerable' due to existing threats.

One of the most important and influential ecological features in the vicinity of the Project is the Falémé River and its associated tributaries. The reach of the Falémé River directly downstream of the Project is not within any protected areas and is not protected itself. Seasonally inundated wetlands are located in the Project Development Area and surrounds, as well as along the Falémé River.

The main terrestrial habitat types within the Project Development Area are tree savannah, shrub savannah, woody savannah, gallery forest, and bowal. All habitat types have moderate levels of

disturbance. Signs of disturbance include land use changes (e.g. cultivation and grazing), timber harvesting, uncontrolled burning and artisanal mining. Some areas of gallery forest in the north of the Project Development Area have been highly disturbed or cleared, while patches of gallery forest along the Falémé River are relatively intact.

In addition to several regionally important species, the baseline field studies confirmed the presence of globally threatened Hooded vulture (Necrosyrtes monachus – Endangered) and Hippopotamus (Hippopotamus amphibious – Vulnerable). Three globally vulnerable plants were also identified in the Project Development Area during field surveys: Afzelia africana, Khaya senegalensis, and Vitellaria paradoxa.

In addition, two globally threatened large mammals were reported by local villagers as being occasionally present in the vicinity of the Project area, although their presence has not been confirmed:

- Chimpanzee (Pan troglodytes – Endangered) / West African chimpanzee sub-species P. t. verus – Endangered).
- African lion (Panthera leo – Vulnerable, West African subpopulation classed as Endangered).

No globally threatened aquatic flora or fauna was identified within the Falémé River in the initial dry season surveys conducted; however there was high fish diversity recorded indicating that the river is in relatively good ecological condition. Along with many phytoplankton, zooplankton and macro invertebrates, the Nile crocodile (Crocodylus niloticus), and two endemic fish occur in the Falémé River near the Project Development Area: the freshwater rat-tail (Gymnarchus niloticus) and the African electric catfish (Malapterurus electricus).

20.4.3 Social Setting

The rural landscape surrounding the Project area is sparsely populated and mostly undeveloped. Kéniéba, the nearest sizeable town and the administrative headquarters of the prefecture, is located approximately 40 km north of the Project. The Project is situated within the Kéniéba gold belt, which contains a dense concentration of mineral exploration licences and several operational gold mines including Randgold's Gounkoto and Loulo Projects and Endeavour Mining's Tabakoto and Segala Projects.

There are 22 settlements identified within an approximately 10 km of the Project area, including along the mine access road. Notably one large village (Fadougou) with a total of 1,355 people, and 255 households (per 2012 census), is located within 1 km of the Project infrastructure. It has been observed that recent in-migration to this village is currently resulting in population increases. Consistent with other parts of the region, the Malinke are the dominant ethnic group in Fadougou Village, comprising more than 85% of the population. The Malinke language is the most common language in Fadougou, spoken by 47% of households, followed by Bambara (28% of households).

The major economic/livelihood activities in the communities surrounding the Project are subsistence agriculture (farming and livestock) and artisanal mining (including some semi-industrial scale operations). Currently thirteen artisanal mining sites have been identified in the permit area,

ten of which are active. Internal migration is common in Mali, particularly in the gold mining areas of south-western Mali.

Approximately 44% of the population in Fadougou Village are below the age of 20 years old and 49% of the population were between 21 to 46 years old, representing 93.3% of the population (2012). The primary schooling rate for Kéniéba is 61%, whereby the schooling rate of the Fadougou Village and surrounding villages is reported at 75% for the 2012 to 2013 year. There are primary schools in Fadougou and Médinandi however there are no secondary schools close to the Project.

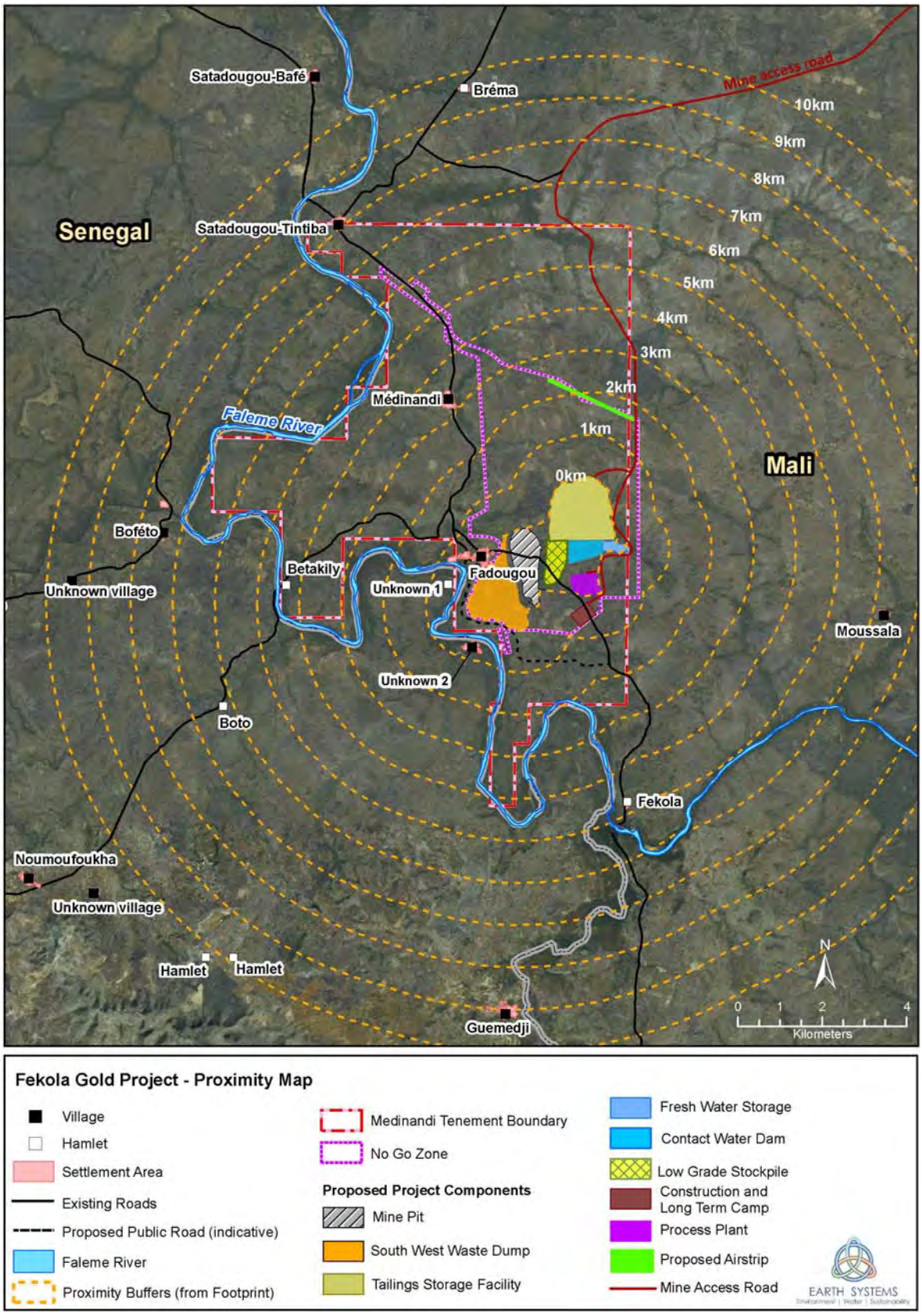
Health infrastructure in the vicinity of the Project is limited. The nearest hospital to the Project is located in Kéniéba approximately 40 km away. Fadougou has one Community Health Centre, one private clinic / infirmary and one pharmacy, while among the closest villages and hamlets, only Médinandi also has a private clinic. Malaria is common in the Kéniéba District. There are also other water and sanitation related illnesses due to poor sanitation and waste management standards. Nutritional issues contribute to health problems due to a shortage in some food supplies in the local area.

Authorities stated that the current water supplies to the local villages surrounding the Project are insufficient for the population, with three manual pump bores in Fadougou, and one in each of the other three closest villages. No electricity-related generation or distribution infrastructure is located in the local villages apart from privately owned diesel generators used only for mining operations. Firewood is the most common form of energy used in the area for cooking and lighting.

Accessibility in the area is generally poor, particularly in the wet season. Unformed dirt tracks connect villages and hamlets in the local area, and provide access to Kéniéba. These are important transport routes for residents for employment, education and livelihood purposes; including the closest markets in Kéniéba or Kolomba. Two mobile telephone networks are available in most surrounding villages however internet is not accessible in the local villages.

There are no known sites of international or national archaeological or cultural heritage significance within the Project Area or surrounds. However, surveys identified 34 archaeological and cultural sites within the broad exploration permit area (Médinandi Tenement), including ancient habitats, scree mounds, iron reduction sites, worship sites, memorial sites, and a place of refuge. Several of these are located in the direct vicinity of the Project footprint.

Figure 20.4.3 Proximity Map of Villages in the Area Surrounding the Project



20.5 Summary of Key Environmental and Social Potential Impacts

A detailed assessment of potential environmental and social impacts from the development of the Project was conducted. The methodology employed utilises a systematic and repeatable process of identifying the potential impacts arising from the development and rating their significance. The potential impacts from the development of the Project were evaluated for the construction, operations, and decommissioning / post-closure phases. The methodology for assessing the impact significance for each aspect includes the assessment of the receiving resource / receptor sensitivity and/or value, as well as evaluation of the impact magnitude associated with the proposed Project development under normal operating conditions, in order to determine an overall impact significance rating.

Potential risks associated with the proposed Project development were also assessed and discussed, as part of the impact assessment process. The assessment and discussion of potential risks associated with the Project development was broadly aligned with internationally accepted risk assessment methodologies, such as ISO31000 Risk management — Principles and Guidelines (2009) and ISO31010 Risk Management – Risk Assessment Techniques (2009).

The approach in the ESIA Update has been to build on the existing assessment work conducted for the 2013 ESIS and to update the impact assessment to reflect modifications to the updated Project design and to align the assessment with relevant international standards.

Wherever possible, environmental and social impacts have been iteratively avoided or reduced throughout the ESIA Update process by making changes to the Project design (e.g. movement of a road to avoid a key environmental feature). A key principle for the implementation of mitigation measures to reduce potential impacts is the mitigation hierarchy. As per IFC Performance Standard 1, adopting the mitigation hierarchy is to "*to anticipate and avoid, or where avoidance is not possible, minimize, and, where residual impacts remain, compensate / offset for risks and impacts to workers, affected communities, and the environment*." The application of the mitigation hierarchy is to implement a set of prioritised steps to avoid and minimise the potential impacts of the Project to the extent practical, and compensate for residual impacts. The highest priority action is avoidance, while the lowest priority is providing offsets.

Potential impacts were assessed for the various aspects characterized in the environmental and social baseline investigations discussed in Sections 20.4 and 20.5. Following the implementation of proposed mitigation measures and under normal operating conditions, identified potential impacts are not estimated to cause significant long-term, adverse impacts on receptors / the receiving environment. Key findings from the impact assessment included the following areas:

- Air quality.
- Water.
- Biodiversity.
- Access to land resources.
- Livelihood and employment.

- Social services / infrastructure.

Key findings related to these aspects are summarized briefly in the following sub-sections.

20.5.1 Air quality

The proposed Project operations will result in fugitive particulate (i.e., dust) emissions (Total Suspended Particulates (TSP), PM_{10} and $PM_{2.5}$) and gaseous emissions from generators and vehicle exhaust pipes (diesel particulate matter, NO_2 and SO_2), with the key potential impact to receptors being fugitive particulate emissions.

The main source of particulate emissions at the Project is expected to be vehicle entrainment, with crushing, drilling and blasting, and generator exhaust emissions contributing to a lesser extent. Impacts are expected to varying degrees to the surrounding flora and fauna and in the five communities in the immediate Project area; however, the greatest impacts will occur at the village of Fadougou due to its proximity. Without mitigation, adverse impacts from fugitive particulates at the village of Fadougou could be significant with particulate concentrations at receptors expected to exceed relevant guidelines.

Mitigation Measures and Residual Impacts

B2Gold will implement a number of mitigation measures to reduce fugitive particulate concentrations at sensitive receptors (e.g., village of Fadougou). Typical mitigation measures to be implemented include:

- Location of facilities / equipment that produce emissions away from sensitive receptors.
- Installation of filters, baghouses, screens, etc. where practical.
- Water sprays on haul roads.
- Chemical suppression on haul roads.
- Water sprays at crushers.
- Water sprays at materials handling points.
- Water sprays on drill rigs.

In addition, air quality monitoring of fugitive particulate and gaseous emissions will be conducted throughout the life of the Project, and additional mitigation will be implemented if necessary to ensure compliance with relevant guidelines. With the implementation of proposed mitigation measures, it is expected that impacts at the village of Fadougou may still be moderated but should be reduced to acceptable levels. Post-closure, following the cessation of the majority of fugitive dust source activities and rehabilitation and revegetation of the Project site, the residual long-term impacts from fugitive particulate emissions are expected to not be significant.

20.5.2 Hydrology, Hydrogeology and Water Quality

Potential hydrology, hydrogeology and water quality impacts of the Project were initially considered in the 2013 ESIS. Since the preparation of the 2013 ESIS, design modifications such as locating the majority of Project infrastructure in a single catchment and removing the requirement for abstraction of water from the Falémé River have resulted in a significant reduction in the potential hydrology, hydrogeology, and water quality impacts of the Project.

Hydrology

Impacts to the local surface water hydrology during construction are expected to be minor to moderate. The morphology of Drainage 3 (Figure 20.4.1), an ephemeral creek which flows seasonally through the main Project development area, will be altered with diversion of surface water around the northern extent of the pit footprint to the catchment of another creek to the north (Drainage 2). Subsequently, the flow in Drainage 3 downstream of the Project Footprint will significantly decrease with impoundment and diversion of the upper reach of this ephemeral creek.

During operations, impacts to the local surface water hydrology are expected to be minor to moderate. Three ephemeral creeks in the vicinity of the Project (Drainages 2, 3, and 4) will be impacted during the rainy season. As with the construction phase, a significant decrease in flows for Drainage 3 is expected downstream of the Project footprint with the diversion of water around the lower reaches of this catchment (to Drainage 2 in early operations and the lower reaches of Drainage 4 for the remainder of the operations phase), and capture of surface water run-off within the TSF, Contact Water Storage Pond, and Fresh Water Storage Pond. The Fresh Water Storage Pond is expected to release water throughout the year, while dewatering of the open pit is conducted, and the Contact Water Storage Pond is expected to discharge, water quality permitting, during the wet season. This is expected to result in flows in the Project diversion channel, the lower reaches of Drainage 4 and the release of water to the Falémé River during mining.

Impacts to regional hydrology (e.g. the Falémé River) are expected to be negligible during both construction and operations as water abstraction from the Falémé River is not expected to be required and Drainage 3 represents less than 0.1% of the total catchment for the Falémé River at its point of intersection (i.e. at Fadougou). Additional flows, to be released to the Falémé River from either of the Storage Water Ponds, are expected to be negligible compared with the median seasonal flow of the Falémé River (i.e. mostly less than 1% of the Falémé River flow). In the event that water abstraction from the Falémé River is required to supplement Project water supply, a minimum environmental flow will need to be established, below which the Project will not abstract water, in order to protect downstream water use and base aquatic ecology.

Post-closure impacts to local surface water hydrology in the Project Area are considered minor, with primarily only the lower reaches of one creek (Drainage 3) affected.

Water Quality

The primary impact to surface water quality during construction will be suspended sediments generated from land clearing / earthworks, extraction from quarries/borrow areas, and road construction / unsealed road surfaces. The majority of input will result from water erosion of disturbed areas during the wet season. Water courses most affected by suspended sediments are

likely to be Drainage 3 and, to lesser extent, Drainages 2, 4 and the Falémé River. Overall, water quality impacts during construction are expected to be moderate.

Progressive rehabilitation and established erosion and sediment control measures are expected to reduce suspended sediment water quality impacts during the operations phase. Static geochemical testwork indicates that the geochemical risk associated with the waste rock, tailings and pit wall rock is low, with no net generation of acidic drainage expected and leachate expected to contain only low dissolved metals concentrations. Kinetic geochemical testwork is being conducted to further characterise potential geochemical impacts. Process water will not be discharged from site during operations and will be recycled from the TSF supernatant pond to the Process Plant. Cyanide concentrations in the TSF supernatant pond will be maintained below 50 mg/L weak acid dissociable (WAD) cyanide via cyanide destruction (INCO Process) in the tailings liquor, throughout the operational life of the Project in order to protect wildlife (e.g. birdlife attracted to a water body) and livestock that may access the TSF. Surface water run-off from the pit, waste rock dumps (if water quality requires), low grade ore stockpile and process plant will be managed through the Contact Water Storage Pond. The Fresh Water Storage Pond will capture upstream surface water run-off and pit dewatering from the dewatering bores. The Contact and Fresh Water Storage Ponds are expected to routinely discharge water to the receiving environment during the wet season. Water quality will be routinely monitored and water will only be discharged once the applicable discharge standards are met.

Rehabilitation and established erosion and sediment control measures are expected to reduce suspended sediment water quality impacts post-closure. However, there may be increased suspended sediment concentrations in downstream water quality at decommissioning, associated with rehabilitation earthworks. One-off treatment of water bodies (e.g. TSF supernatant pond, Contact Water Storage Pond and Fresh Water Storage Pond), if required, prior to discharge to the environment at decommissioning is expected to minimise potential water quality impacts associated with the release of these waters.

Groundwater

A groundwater cone of depression, surrounding the open pit, is expected to develop during operations, associated with the dewatering of the pit to allow for mining operations below the groundwater table. The extent of the groundwater cone of depression is expected to be greater than 2.5 km, from the edge of the pit, in a north-south direction. The village wells for Fadougou and Médinandi are within the zone of groundwater level drawdown, and groundwater levels in the vicinity of these two villages are expected to decrease by greater than 10 m. Groundwater level drawdown will be monitored during operations and alternative water supplies identified and provided for Fadougou and Médinandi villages in the event that water supply from the village wells is interrupted.

Rehabilitation of the TSF at decommissioning, including treatment of the TSF supernatant pond at decommissioning, is expected to minimise the potential for residual impacts. The pit is expected to act as a sink for any seepage from the TSF. Estimates of post-closure pit lake water quality indicate that water will meet relevant water quality standards. Monitoring of surface and groundwater post-closure will be implemented to determine whether additional measures are required to avoid post-closure impacts to water quality.

Post-closure, groundwater levels are expected to rebound after the cessation of pit dewatering. Groundwater levels are expected to take approximately 55 years after closure to reach equilibrium as the pit lake reaches its capacity. Groundwater level impacts post-closure are expected to be far lower than during operations. However, residual groundwater level impacts post-closure will need to be quantified during operations.

With the implementation of the detailed water management and monitoring program proposed for the Project (refer ESIA Update Volume II) the potential water quality and hydrology impacts and risks from the Project and subsequent impacts on water use will be greatly reduced. The Project will also implement a number of significant measures to ensure any unavoidable impacts of the Project on water use are compensated for. These could include a household latrine building programme, developing an institutional support programme to establish or strengthen community hygiene committees in Project affected villages, and construction of potable water infrastructure in key villages.

20.5.3 Biodiversity

A number of priority biodiversity features (including threatened and protected species) that will be impacted by the development of the Project have been identified in the Project area. There are no identified Protected Areas in the vicinity of the Project. Key potential adverse impacts to biodiversity are presented in the following sub-sections.

Terrestrial Flora

The most significant adverse potential impacts on terrestrial habitat and flora as a result of the Project are expected to be:

- Habitat loss and associated degradation.
- Potential loss of individual stems of priority flora.
- Over exploitation of natural resources and poaching due to in-migration.

Terrestrial Fauna

The principal potential impacts on terrestrial fauna as a result of the Project are expected to be:

- Habitat loss and associated degradation – natural habitats will be removed for Project components and roads.
- Barriers to movement from the clearance of habitat and construction of the Project components.
- Noise and vibration.
- Vehicle collisions.
- Drinking contaminated water (e.g. from the TSF).

- Light spills.

Aquatic Habitats and Fauna

The principal potential impact to aquatic biodiversity for the Project will relate to potential physical impacts on hydrology and water quality. Other potentially significant impacts will include direct habitat loss and degradation associated with the siting of the Project Footprint, and increased pressure on aquatic resources from Project-related population increases.

Mitigation Measures and Residual Impacts

Throughout the Project, B2Gold will implement numerous mitigation measures including commitments identified in the 2013 ESIS, requirements of the existing Environmental and Biodiversity Performance Standards (2014) relevant to all B2Gold operational mines, and others. A summary of key mitigation measures is presented as follows:

- A Biodiversity Management Plan will be implemented, communicated, adhered to and maintained.
- Additional, targeted ecological studies of terrestrial flora and fauna and aquatic habitats and fauna will be conducted to increase knowledge about species presence in the area.
- Environmental awareness raising and training will be conducted aimed at minimising the impacts on flora and fauna.
- Noise generating activities will be mitigated to be within legal noise limits and disturbance levels will be kept to a minimum and restricted to operational areas.
- The design, construction and operation of facilities will be focused on the lowest level of intrusion, through the use of silencers, non-reflective non-toxic paints, low wattage / coloured lighting pointing towards infrastructure and away from the natural surroundings, and the use of vegetation as buffers around infrastructure.
- Water storage facilities, diversion structures and containment facilities will be designed to limit and reduce fauna access and to provide easy egress for any trapped fauna.
- Eradication programs will be considered with regards to pest flora and fauna species.
- Where the operation of mobile machinery can impact wildlife, control measures shall be developed, implemented and monitored to reduce injury or mortality. Control measures could include fencing of impacted areas or lease boundaries, reducing speed limits, installation of wildlife corridors and/or training and awareness of employees and contractors.
- Site procedures will be developed, implemented and communicated to prohibit:
  - unauthorised hunting, trapping or fishing or the unauthorized harvesting of forest products

- minimising and/or restricting access to designated areas.

- Prior to vegetation clearance, 'animal scaring' activities will be conducted to reduce and minimize loss of fauna individuals during the removal of vegetation. This will include rescue of vulnerable individuals, such as females with cubs or nests with chicks, and their transfer to suitable sites for further release.

- Biodiversity monitoring programs will be developed, implemented and periodically reviewed. These monitoring programs will include the monitoring of any potential or actual adverse biodiversity impacts (e.g. sensitive or protected areas) and the effectiveness of implemented control measures.

- The measures included in the Rehabilitation and Preliminary Mine Closure Plan will be implemented at the Project site. The site will be rehabilitated and revegetated concurrently during operations to the extent practical and fully at closure to restore impacted flora and habitats and provide self-sustaining natural ecosystems within the Project Development Area.

In addition to the measures described above, given the potential impacts of the Project on a number of priority biodiversity features, B2Gold will consider providing biodiversity offsets for the Project. The requirement for biodiversity offsets will be assessed based on the outcomes of the additional ecological surveys to be conducted for the Project. The Biodiversity Action Plan provides a framework for the development of a detailed Biodiversity Offset Strategy that would need to be developed if biodiversity offsets were to be implemented for the Project.

Potential impacts to biodiversity during construction and operations area expected to be moderate, however, with the implementation of proposed mitigation measures including the rehabilitation and revegetation of the Project site following mine closure the expected residual long-term impacts are expected to not be significant.

20.5.4 Access to Land Resources

The communities in the Project area rely on land resources to make a subsistence living, primarily via small scale farming and/or artisanal mining activities. The community most significantly impacted is the village of Fadougou. The Project's No-go Zone has isolated the village and reduced livelihood options for the community. In particular, the No-go Zone which:

- Leaves the village wedged between the No-go Zone to the north-east and the Falémé River to the west, blocking expansion to the east, north-east and south-east.

- Passes within 50 m of some village structures in the school sector (east) and community health sector (south-east).

- Will block access to agricultural properties located east of the mine.

- Has required the relocation of some properties to new locations northwest of Fadougou. Good land is now a scarce resource.

- Has resulted in some Fadougou villagers occupying land owned by the Médinandi village to the north with resulting potential for future conflict.

- Restricts use of access roads connecting Fadougou-Moussala, Fadougou-Fadougou Niafa, and Fadougou-Guémedjè.

As a result of these impacts, the village of Fadougou, at its present location, will face significant impacts relating to land resource access that will worsen if the population increases from the influx of people expected in the advent of an operating mine.

B2Gold will implement various mitigation measures to reduce the impacts to land resources. These measures are described in the Community Development Plan (CDP) developed for the Project (see CDP Section 20.7.10). The overall objective of the CDP is to create an enhanced, sustainable economy following mine closure. Specifically relating to land resources the CDP outlines the framework by which the Project can work with local communities to improve land resource productivity (i.e., agriculture, livestock, and forest resources) and promote other sustainable economic drivers to compensate for potential losses or negative impacts.

The above mentioned potential impacts to land resources can be mitigated and reduced to acceptable levels. However, residual impacts will likely remain following the implementation of such measures resulting in the potential for complaints by the community and its inhabitants and conflicts between the community and its inhabitants and the Project. Due to this potential for conflict, B2Gold will evaluate resettlement of the village of Fadougou (described in the following sub-section).

Potential Resettlement of Village of Fadougou

Although resettlement of the village of Fadougou is not necessary, resettlement can ensure access to adequate land resources now and in the future. In addition, resettlement would eliminate or minimize other potential social, safety or environmental risks to the community (e.g., eliminate potential safety risks such as fly rock and air blast, greatly reduce the risk of vehicle and equipment accidents, and minimize potential environmental impacts (e.g., fugitive dust, noise) to negligible levels.

If it is decided that resettlement of the village of Fadougou is beneficial to eliminate or reduce negative impacts and maximize beneficial impacts to the community, resettlement would be implemented in compliance with Malian law and in line with international best practice, i.e., International Finance Corporation (IFC) Performance Standard 5: Land Acquisition and Involuntary Resettlement. When resettlement is not able to be avoided, IFC Performance Standard 5 objectives include:

- The minimization of adverse social and economic impacts from land acquisition or restrictions on land use by:

  - providing compensation for loss of assets at replacement cost

  - ensuring that resettlement activities are implemented with appropriate disclosure of information, consultation, and the informed participation of those affected.

- Improving or restoring the livelihoods and standards of living of displaced persons.

- Improving living conditions among physically displaced persons through the provision of adequate housing with security of tenure at resettlement sites.

Preliminary construction activities have already commenced at the Project site. If determined necessary, resettlement would occur as soon as is practical to reduce potential impacts and conflicts to the greatest extent possible. In general, the potential impacts to Fadougou and its inhabitants are more significant closer to the Project and become less significant as one moves away from the Project, particularly potential environmental impacts such as fugitive dust or noise. Resettlement, therefore, would begin with those homes and inhabitants potentially most impacted by the Project, i.e., those homes and inhabitants closest to the Project. The closest homes and inhabitants would be resettled first, and once resettled, the next closest homes would be resettled and etc. until all of the homes and inhabitants of Fadougou are resettled.

20.5.5 Livelihood and Employment

The development of the Project will have a significant impact on the potential for paid employment and other forms of livelihood in the Project area, via both direct employment by the Project and jobs created in the communities due to the overall increased economic activity in the area. In general, these impacts are positive, however, the local residents in the communities surrounding the mine have little formal work experience, a general low level of relevant skills, and there will not be sufficient available jobs to employ all the locals in search of work. If not managed properly, it is possible that the potential positive impacts of employment will not be fully appreciated in the Project area. To this extent B2Gold will implement a number of measures to ensure potential positive impacts are maximized throughout the life of the mine, including:

- B2Gold will preferentially hire employees based on the following criteria:

  - Malian nationals, who possess the required skills and abilities

  - candidates over 18 years old, and

  - current local residents and returning local residents, in preference over job seekers who have recently arrived in the area.

- B2Gold has committed during public consultation with the local communities to hire local residents to fill at least 60% of the unskilled job positions at the mine site.

- B2Gold has committed to establishing a training centre focused on increasing skills of youth in the communities surrounding the Project to prepare them for potential employment with the mine during operations.

In addition to measures aimed at maximizing direct employment positive impacts in the Project area, B2Gold will also implement initiatives in the surrounding communities to create sustainable small- and medium-scale enterprise and other sustainable economic programs throughout the life of the Project. A Community Development Plan (CDP) has been developed that presents the framework to promote sustainable community development opportunities in the Project area (see CDP Section 20.7.10).

Following closure of the Project, economic activity and direct employment will most likely fall compared to operations. There is the potential that this reduction in activity leads to negative impacts associated with unemployment and loss of business, however, with successful implementation of sustainable community development initiatives throughout the life of the Project and additional community development initiatives specifically targeted at the post-closure period it is expected that potential negative long-term impacts can be minimized and that the overall impact of the Project will be positive.

20.5.6 Social Services / Infrastructure

The population of the five communities in the Project area was estimated in 2012 to be 2,458 people. The population has likely increased since 2012 and will increase further with development of the Project. In general, the communities in the Project area have limited education, health care, sanitation, and etc. services, as well as poor infrastructure (roads, access to electricity, etc.) and not a high level of available goods and services. The expected in-migration due to the development of the Project and its subsequent impacts on the communities in the Project area have the potential to be significant.

In-ward migration can have positive and negative impacts such as:

- Stimulating the local economy and community organization. Wages and salaries will be injected into the local economy creating additional jobs and economic opportunities.
- Developing informal settlements which increase demand / need for basic infrastructure – land, housing, clean water supplies, sanitation, and electricity and waste management systems.
- Increasing pressure on government services such as health and education facilities, their staffing, and capital and operating cost requirements.
- Increasing incidence of social ills including alcoholism, drug abuse, prostitution, and criminality.
- Increasing incidence of disease such as sexually transmitted diseases including HIV / AIDS and tuberculosis.

The above impacts can have severe negative consequences that can last beyond the life of the Project. To mitigate these potential negative impacts and ensure that the communities in the Project area have an enhanced, sustainable economy following mine closure, B2Gold will invest in improving infrastructure, education, health care, and other economic programs to promote sustainability. The Community Development Plan (see CDP Section 20.7.10) presents the

framework to promote sustainable community development opportunities in the Project area. To be successful these efforts must be carefully co-ordinated with those of local government and the community through planned engagement and public-private partnerships. B2Gold will engage with local communities and other stakeholders through the measures outlined in the Stakeholder Engagement Plan (SEP) (see SEP Section 20.7.9), including identifying Project stakeholders, ensuring timely and effective communication between the Project site and affected stakeholders throughout the life of the Project, documenting meaningful Project consultation and disclosure, establishing a formal grievance process, and monitoring and reporting upon the performance of stakeholder engagement processes.

With the successful implementation of the measures outlined in the CDP and SEP, is expected that potential negative long-term impacts on social services and infrastructure can be minimized and that the overall impact of the Project will be positive.

20.6 Summary of Environmental and Social Management and Monitoring Plan

An Environmental and Social Management Plan was developed as part of the 2013 ESIS that documented the proposed mitigation and control measures designed to meet Project regulatory requirements and reduce potential impacts from the proposed Project to acceptable levels. This Environmental and Social Management Plan has been updated to account for optimization and modification of the Project and a corresponding Environmental and Social Management and Monitoring Plan (ESMMP) has been developed as part of the ESIA Update. This ESMMP is supported by a number of individual Management Plans (MP's) that describe how the site will meet relevant regulations, standards and guidelines and manage and minimize key environmental and social risks of the Project. The ESMMP and its supporting individual MP's are "living documents" which will be amended periodically throughout the life of the Project to reflect changes in procedures, practices, Project phase, etc.

The following Management Plans have been created:

- Environmental Monitoring Plan.
- Water Management Plan.
- Waste Management Plan.
- Erosion and Sediment Control Plan.
- Rehabilitation and Preliminary Mine Closure Plan.
- Emergency Preparedness and Response Plan.
- Biodiversity Action Plan.
- Stakeholder Engagement Plan.
- Community Development Plan.

Additional MP's will be created as the Project develops and as they become necessary.

20.6.1 Environmental Monitoring Plan

The Environmental Monitoring Plan (EMP) will continue to build upon the environmental database that was generated to characterize baseline conditions for the 2013 ESIS and the 2015 ESIA Update. Ongoing environmental monitoring programs will continue to document site conditions, focusing on the generation of data sets that will allow evaluation of the environmental performance of specific project components. The EMP covers data collection of the following environmental aspects:

- Meteorology.
- Air quality.
- Noise and vibration.
- Water.
- Biodiversity.

20.6.2 Water Management Plan

There are a number of sources in all Project phases that have the potential to adversely impact water resources. The Water Management Plan (Water MP) outlines the strategies, operational controls and management practices that will be put in place to effectively manage water at the Project site. The Water MP includes water quality objectives, describes site water balances, and defines process water, stormwater, discharges and dewatering activities to ensure that human health and the environment are protected.

20.6.3 Waste Management Plan

Waste is generated during all phases of the Project. Waste streams likely to be generated during the life of the Project include the following:

- Mining / Processing wastes (waste rock and process tailings).
- Hazardous wastes (waste oils, solvents, laboratory wastes, and medical wastes).
- Domestic wastes (inert wastes such as plastic, glass, and construction materials).
- Organic wastes (food and plant material).

The Waste Management Plan (Waste MP) details the handling, storing, and disposing of non-process wastes (i.e., excluding waste rock, ore and tailings) that are generated by Project activities, including the mining operations, Process Plant, workshop, offices, tailings storage facility (TSF) and other facilities. The Waste MP also addresses the management of wastes designated for special

handling including hazardous, medical, and emergency response wastes and sewage as well as the minimization of waste through composting and recycling programs.

20.6.4 Erosion and Sediment Control Plan

Project development, and in particular Project construction, will expose soils to erosive forces of water through precipitation and stormwater runoff. The Erosion and Sediment Control Management Plan (ESCMP) documents B2Gold's approach to avoiding and / or mitigation of erosion and sediment transport during the life of mine and post-closure for the Project. The ESCMP documents general ESC Best Management Practices (BMPs) that will be considered for implementation pending site-specific requirements and provides recommendations for site-specific measures for Project facilities prone to erosion or near sensitive receptors.

20.6.5 Rehabilitation and Preliminary Mine Closure Plan

A Rehabilitation and Preliminary Mine Closure Plan (RMCP) has been developed as the first step in providing a clear methodology for progressive rehabilitation and planning for decommissioning and closure the Project. Additional details regarding the RMCP are provided in Section 20.7.

20.6.6 Emergency Preparedness and Response Plan

The Emergency Preparedness and Response Plan(EPRP) outlines the processes to be implemented at the Project for responding to on-site emergencies, including accidental spills and releases, fires, explosions, medical emergencies and others. It is designed to minimize employee exposure to risk and injury and limit potential impacts to the environment in emergency situations. The EPRP identifies training programs for emergency recognition and prevention, methods and procedures for alerting on-site employees, resources for medical and emergency treatments, and procedures for containment and clean-up of accidental spills.

20.6.7 Biodiversity Action Plan

A number of priority species were identified in the vicinity of the Project that may potentially be impacted by the development of the Project. The objective of the Biodiversity Action Plan (BAP) is to avoid and minimise potential residual impacts on Project area biodiversity wherever feasible. The BAP outlines specific mitigation measures to be implemented at the site and outlines further studies required to reduce current ecological uncertainties associated with threatened flora and fauna at the Project site. The BAP also provides a framework for the potential development of a biodiversity offset strategy to compensate for residual impacts of the Project and allow B2Gold to maintain (i.e. no net loss) or increase Project area biodiversity.

20.6.8 Stakeholder Engagement Plan

Experience has demonstrated that projects that establish early and ongoing stakeholder engagement processes that continue throughout the life of the project have a greater rate of success, as well as reduced business risk. B2Gold has prepared a Stakeholder Engagement Plan (SEP) that describes how it will collect and disseminate information to people who are affected by and/or interested in the Project. The SEP establishes the basis for how B2Gold will conduct its business with the community to ensure transparency in Project activities as they may affect

stakeholders. The SEP describes specific measures to identify Project stakeholders, ensure timely and effective communication between the Project site and affected stakeholders throughout the life of the project, document meaningful Project consultation and disclosure, establish a formal grievance process, and monitor and report upon the performance of stakeholder engagement processes.

20.6.9 Community Development Plan

B2Gold recognizes that a community development program is essential in ensuring that local communities do not become dependent on the mining economy, but rather have an enhanced, sustainable economy following mine closure. B2Gold will invest in improving infrastructure, education, health care, and other economic programs to promote sustainability. The Community Development Plan (CDP) presents the framework to promote sustainable community development opportunities in the Project area. The CDP also describes measures for collecting stakeholder input through public consultation and for planning, development, implementation, and monitoring of the community development programs.

20.7 Mine Rehabilitation and Closure

B2Gold's overall objective for mine closure is to prevent or minimise adverse long-term environmental, physical, social and economic impacts for the greater Project area and to create stable landforms that provide self-sustaining natural ecosystems within the Project Development Area. A *Rehabilitation and Preliminary Mine Closure Plan* (RMCP) has been developed as the first step in providing a clear methodology for progressive rehabilitation and planning for decommissioning and closure the Project. The aim of the RMCP is to provide a rehabilitation and closure framework that will be refined throughout the life of the Project to ultimately achieve mine closure.

Management and mitigation measures specified in the Fekola Gold Project ESIA, Environmental and Social Management and Monitoring Plan (ESMMP), and Biodiversity Action Plan (BAP) will be implemented to avoid or minimise potential environmental and social impacts. However, potential residual impacts at the completion of mining and processing operations will require evaluation and subsequent actions with regard to: safety and hazards risks, potential sources of ongoing pollution, future land use and resource demands, ecological compatibility, community expectations, aesthetics, and cost.

Ongoing refinement of the RMCP will require community consultation regarding intended end post-mining land use for the Project Development Area and development of the methodology for clearly defining success or failure of rehabilitation (completion criteria). In subsequent versions of the RMCP, B2Gold will develop a consistent approach to performance monitoring and identify maintenance and/or remedial measures required for rehabilitation that promote the achievement of completion criteria to avoid or minimise residual impacts to the satisfaction of Project stakeholders.

A preliminary closure cost estimate has been developed. Costs were provided by B2Gold and its consultants including Lycopodium and Knight Piésold. Where costs were not provided, supplemental costs were utilitized from other sources including similar mining projects. Closure costs were estimated at full mine build-out. Concurrent rehabilitation will be conducted during the life of the mine to the extent practical to minimize impacts and lower the environmental liability at

closure, however, the bulk of the rehabilitation and closure costs are not expected to be incurred until the cease of operations. If the mine were to close prior to the expected end of mine life the bulk of the closure costs would be incurred at this time. The estimated rehabilitation and closure costs for the Project are approximately US$ 20.4M over the life of the mine. These are summarized in Table 20.7.1 below.

Table 20.7.1 Summary of Estimated Rehabilitation and Closure Costs

Component	Total*
Physical Closure Costs	
Pit	250,000
Ore Processing	2,780,000
Waste Rock Dump	2,800,000
Water Management	490,000
Tailings Storage Facility	4,230,000
Other Buildings and Infrastructure	1,500,000
Closure Maintenance (assume 10% area)	1,460,000
Total Physical Closure	**$13,500,000**
Other Closure Costs	
Planning, Auditing and Reporting	20,000
Post-Closure Monitoring	1,430,000
Management / Administration / Personnel (10%)	1,350,000
Engineering / Design / Permitting (10%)	1,350,000
Contingency (20%)	2,700,000
Total Other Closure	**6,850,000**
Total Closure Costs	**20,400,000**

***Totals may not be exact due to rounding**

FEKOLA GOLD PROJECT

NI 43-101 - CAPITAL AND OPERATING COSTS

3174-STY-002

Table of Contents

			Page
21.0	**CAPITAL AND OPERATING COSTS**		**21.1**
	21.1	Operating Costs	21.1
		21.1.1 Mining Costs	21.1
		21.1.2 Plant and Administration Costs	21.1
		21.1.3 Power	21.2
		21.1.4 Operating Consumables	21.2
		21.1.5 Labour (Processing / Maintenance and Administration)	21.3
		21.1.6 General and Administration Cost (excluding G&A labour)	21.3
		21.1.7 Maintenance	21.4
		21.1.8 Exclusions	21.4
	21.2	Capital Cost Estimate	21.5
		21.2.1 Summary	21.5
		21.2.2 Estimating Currency and Base Date	21.6
		21.2.3 Mining Capital Costs	21.7
		21.2.4 Estimating Methodology	21.8
		21.2.5 Quantity Development	21.8
		21.2.6 Pricing Basis	21.9
		21.2.7 Field Indirects	21.11
		21.2.8 Engineering Services	21.12
		21.2.9 Owner's Costs	21.12
		21.2.10 Spares	21.12
		21.2.11 Duties, taxes and Insurances	21.13
		21.2.12 Contingency	21.13
		21.2.13 Deferred Capital	21.14
		21.2.14 Escalation	21.14
		21.2.15 Qualifications and Assumptions	21.14
		21.2.16 Exclusions	21.14
	21.3	Deferred Capital	21.15
	21.4	Project Implementation	21.16
		21.4.1 Implementation Strategy	21.16

TABLES

Table 21.1.1	Fekola Mine Operating Cost Summary	21.1
Table 21.1.2	Fekola Mine Operating Cost per Mining Classification	21.1
Table 21.1.3	Fekola Process Plant 4 Mtpa Operating cost Summary	21.2
Table 21.1.4	Fekola Process Plant Power Cost by Plant Area	21.2
Table 21.1.5	Fekola Process Plant Consumables Cost by Plant Area	21.3
Table 21.1.6	Fekola Processing Plant and Admin Manning Levels	21.3
Table 21.1.7	Fekola Plant General and Administration Summary	21.4
Table 21.1.8	Fekola Plant Maintenance Materials Cost	21.4
Table 21.2.1	Capital Estimate Summary by Area (2Q15, ±15%)	21.6
Table 21.2.2	Capital Estimate Summary by Discipline (2Q15, ±15%)	21.6
Table 21.2.3	Foreign Currency Exposure	21.7
Table 21.2.4	Mining Capital Costs	21.8
Table 21.2.5	Derivation of Quantities	21.9
Table 21.2.6	Sources of Pricing	21.10
Table 21.2.7	Standard Direct Labour Gang Rates	21.11
Table 21.2.8	Contingency Percentage Summary	21.13
Table 21.3.1	Deferred Capital Summary	21.15

Table of Contents (Continued)

Page

Table 21.3.2	Deferred Capital Schedule	21.15
Table 21.4.1	Key Project Milestones	21.16

21.0 CAPITAL AND OPERATING COSTS

21.1 Operating Costs

21.1.1 Mining Costs

MSC prepared the mining operating cost for Fekola based on an owner operator mining strategy. Table 21.1.1 and 21.1.2 summarise the mining operating costs over the Life of Mine (LOM), based on a 4 Mtpa processing plant.

Table 21.1.1 Fekola Mine Operating Cost Summary

Cost Centre	US$ LOM	US$ /t Rock Mined	US$ /t Ore Mined	US$ /oz[1] Au
Base Mining Costs	693,276,000	2.58	14.1	200.9
Additional Ore Costs				
Administration and Supervision	24,437,000	0.09	0.50	7.1
Grade Control	30,743,000	0.11	0.63	8.9
Selective Mining Extra	9,787,000	0.04	0.20	2.8
ROM Loader Operation	20,235,000	0.08	0.41	5.9
Stockpiling and Rehandling	25,108,000	0.09	0.51	7.3
Total	**803,584,000**	**2.99**	**16.34**	**232.92**

Table 21.1.2 Fekola Mine Operating Cost per Mining Classification

Cost Centre	US$ LOM	US$ /t Rock Mined	US$ /t Ore Mined	US$ /oz[1] Au
Load Haul and Ancillary Equipment	473,035,000	1.76	9.62	137.11
Drilling and Blasting	234,490,000	0.87	4.77	67.97
Maintenance Labour and Overheads	65,968,000	0.25	1.34	19.12
Administration and Supervision	30,091,000	0.11	0.61	8.72
Total	**803,584,000**	**2.99**	**16.34**	**232.92**

[1] **Figures based on the average LOM estimate for annual gold production of 276,000 oz per year.**

US$ 11.905M worth of pre-stripping and stockpiling costs have been capitalised and are not included in the above operating cost summary.

21.1.2 Plant and Administration Costs

Process plant and administration operating costs have been developed by Lycopodium and B2Gold based on a design treatment rate of 4 Mtpa of ore with the plant operating 24 h/d, 365 d/y with a 94% plant utilisation (nominal 8,234 h/y) and a P_{80} grind size of 75 µm.

The operating cost estimate has been compiled from a variety of sources and is based on whole of ore treatment and a head grade of 2.5 g/t Au ore.

Operating costs are presented in United States Dollars (US$) and are based on prices obtained during the second quarter of 2015, to an accuracy of ±15%. The process plant operating costs for the facilities are summarised in Table 21.1.3.

Table 21.1.3 Fekola Process Plant 4 Mtpa Operating cost Summary

Cost Centre	US$ M/y	US$ /t Ore	US$ /oz Au[1]
Processing Labour	3,337,124	0.83	12.10
Power	36,807,925	9.20	133.37
Consumables	29,097,026	7.27	105.43
Maintenance Materials	3,933,073	0.98	14.26
Subtotal - Processing & Maintenance	**73,184,522**	**18.29**	**265.16**
General & Administration	**14,893,612**	**3.72**	**53.96**
Total	**88,078,134**	**22.01**	**319.12**

[1] **Figures based on the average LOM estimate for annual gold production of 276,000 oz per year.**

21.1.3 Power

The power requirements for the process plant were based on the mechanical equipment list and adjusted for equipment load factor and utilisation. A power unit price of US$ 0.20/kWh was calculated based on an HFO fuelled on site power station and supported by vendor quotation and stated fuel consumption data. A summary of the power cost for the plant by plant area is tabulated below in Table 21.1.4.

Table 21.1.4 Fekola Process Plant Power Cost by Plant Area

Plant Area	Total Power (US$ /t)
Feed Preparation	0.15
Milling	7.17
Trash Removal / Thickening	0.14
Leaching	0.68
Elution / Goldroom	0.05
Tails Handling	0.40
Reagents	0.03
Water, Plant, & Air Services	0.57
Mining Services	0.02
Total	**9.20**

21.1.4 Operating Consumables

The consumables consumption requirements for the Fekola process plant were based on OMC crushing and milling data, testwork consumption rates and industry standards. An allowance for

wastage has been included. Budget quotations for reagents and consumables were received from suppliers and adjusted to a DAP (Delivered at Place) price and include customs and duties. The diesel cost was supplied by B2Gold and diesel consumption for the plant mobile equipment was estimated. The consumables cost by plant area is summarised below in Table 21.1.5.

Table 21.1.5 Fekola Process Plant Consumables Cost by Plant Area

Plant Area	US$ /t ore
Feed Preparation	0.04
Milling	3.83
Trash Removal / Thickening	0.09
Leaching	1.41
Elution / Goldroom	0.57
Tails Handling	0.90
Water Services	0.12
Laboratory	0.21
Remaining Consumables	0.11
Total	**7.27**

21.1.5 Labour (Processing / Maintenance and Administration)

The labour costs for the process plant and administration are summarised in Table 21.1.6. Labour rates were agreed with B2Gold and are based on information from other West African Projects in the region. The labour rates are based on a skill level and consist of a base salary and the required overhead allowances.

Table 21.1.6 Fekola Processing Plant and Admin Manning Levels

	People	Total Labour Cost (US$ /y)
Administration	150	3,092,087
Operations, Metallurgy and Laboratory	107	2,391,784
Maintenance	43	945,340
Total	**300**	**6,429,211**

The labour roster and manpower requirements were agreed with B2Gold. The roster is based on a four panel roster for shift personnel and 8 hours per shift which is typical for this area.

21.1.6 General and Administration Cost (excluding G&A labour)

General and Administration costs were advised by B2Gold and are based on information from similar West African operations in the region and are summarised in Table 21.1.7 . Administration includes insurance and parasitic power costs.

Table 21.1.7 Fekola Plant General and Administration Summary

General and Administration	US$ /y
Administration Cost	3,579,273
Human Resources	292,245
Information Technology	475,422
Accounting	571,851
Purchasing, Warehousing, Logistics	796,772
Environmental	656,802
Security	794,591
Safety	407,257
Medical Clinic	319,606
Employee Transportation	2,705,881
Cafeteria and Housing	3,412,200
Projects	270,447
Management	611,267
Total	**14,893,612**

21.1.7 Maintenance

The maintenance cost for the Fekola processing plant was factored from the equipment supply capital cost and is summarised in Table 21.1.8. Allowances for plant mobile equipment, and general maintenance have been made.

Table 21.1.8 Fekola Plant Maintenance Materials Cost

	Annual Maintenance Cost US$	Annual Maintenance Cost US$ /t
Total	3,933,073	0.98

21.1.8 Exclusions

The operating costs stated above makes no allowance for the following:

- All B2Gold head office / corporate costs.
- All import duties, withholding taxes and other taxes.
- All sunk costs.
- All project financing costs.
- Any impact of foreign exchange rate fluctuations.

- Any escalation from the date of the estimate, except for the labour unit cost base salary from 2015 to 2017.
- Any contingency allowance.
- Any land compensation costs. All required village compensation/relocations have been previously completed.
- Any rehabilitation or closure costs.
- Any licence fees or royalties.
- Tailings storage costs, including future lifts and rehabilitation. This is captured in Section 21.3 Deferred capital.
- Government monitoring / compliance costs.
- Water extraction licensing costs. (pumping power and installation has been allowed).
- Gold refining costs and transport of gold from site.

21.2 Capital Cost Estimate

The overall capital cost estimate was compiled by Lycopodium and is presented here in summary format. The capital cost estimate reflects the Project scope as described in this report. Mine capital costs (developed by MSC and B2Gold) are included in the estimate tables below.

Knight Piésold provided quantities and rates for the TSF, pipeline corridor, pit diversions, sediment control structures and water storage pond structures which are included in the estimate tables below.

All costs are expressed in US$ unless otherwise stated and based on 2Q 2015 pricing. The estimate is deemed to have an accuracy of ±15%.

The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with B2Gold for scope and accuracy.

21.2.1 Summary

The capital estimate is summarised in Table 21.2.1 by area and 21.2.2 by discipline. The pre-production capital cost is estimated at US$ 395 million.

The estimate Work Breakdown Structure (WBS) is based on the standard Lycopodium Minerals WBS for gold projects.

Table 21.2.1 Capital Estimate Summary by Area (2Q15, ±15%)

Main Area	US$
Construction Indirects	84,200
Treatment Plant Costs	102,195,400
Reagents and Plant Service	23,326,600
Infrastructure	81,954,800
Mining	40,467,300
Engineering	13,206,300
Owners Project Costs	80,865,100
Owners Operation Costs (Working Capital)	6,963,300
Subtotal	**349,063,000**
Contingency	45,937,000
Fees, Taxes & Duties	0
Escalation	0
Grand Total	**395,000,000**

Table 21.2.2 Capital Estimate Summary by Discipline (2Q15, ±15%)

Discipline	Supply Cost	Installation Cost	Ancillary Costs	Contingency	Total
Architectural	6,977,206	2,343,258	-	932,046	10,252,510
Earthworks	2,150,128	26,787,656	-	3,856,718	32,794,502
Concrete	8,989,130	11,251,823	-	2,503,217	22,744,170
Electrical	45,712,081	7,739,621	-	7,725,994	61,711,696
Instrumentation	3,931,932	995,743	-	640,598	5,568,273
Platework	7,825,145	3,891,344	-	1,464,227	13,180,716
Mechanical	52,818,853	6,868,879	-	7,037,950	66,725,682
Piping	10,166,773	8,598,834	-	3,976,019	22,741,626
Steelwork	8,008,935	5,500,593	-	1,676,771	15,186,299
Owners Cost	500,000	22,124,795	70,793,023	11,844,338	105,262,156
Engineering/Consultants	-	-	12,200,000	1,230,000	13,430,000
General	547,654	21,333,315	1,006,280	3,049,123	25,936,372
Grand Total	**147,627,837**	**117,435,861**	**83,999,303**	**45,937,001**	**395,000,000**

The capital cost summary excludes US$ 66.7M in mine fleet and power generation costs which are expected to be lease financed. These costs have been shown in the deferred capital cost schedule in Section 21.3.

21.2.2 Estimating Currency and Base Date

The estimate is expressed in US$ based on prices and market conditions current at second quarter 2015 (2Q 2015).

The following exchange rates have been used in the compilation of the estimate:

- US$ 1.00 = AUD 1.28
- US$ 1.00 = EUR 0.909
- US$ 1.00 = CAD 1.25
- US$ 1.00 = GBP 0.666
- US$ 1.00 = ZAR 12.5
- US$ 1.00 = XOF 606.06
- Foreign currency exposure is shown in Table 21.2.3 below

Table 21.2.3 Foreign Currency Exposure

Currency	Percentage of Capital Estimate
US$	~86%
EUR	~11%
AUD	<3%
GBP	<1%
ZAR	<1%

The estimate capital cost accuracy as defined by the scope in this document and attached appendices is as follows:

- ±15% Plant Capital Costs.
- ±15% Infrastructure Capital Costs.

21.2.3 Mining Capital Costs

Mining capital costs were prepared on the basis of an owner operator mining strategy and include mining activities to the date of plant handover to operations (nominally December 2017). A summary of mining cost included in the capital cost estimate is provided in Table 21.2.4 below.

Table 21.2.4 Mining Capital Costs

Description	US$ (excl. contingency)
Mine Pre-Stripping	10,822,730
Mine Establishment	9,053,040
Mine Pre-Production	2,363,630
Mine Facilities	7,227,900
Mine Mobile Equipment	11,000,000
Total	**40,467,300**

The mining cost summary excludes an additional US$ 49.7 million in mine fleet costs which are expected to be leased financed. These costs have been included in the deferred capital cost schedule in Section 21.3.

21.2.4 Estimating Methodology

General arrangement drawings and a layout 3D model have been produced with sufficient detail to permit the assessment of the engineering quantities for earthworks, concrete, steelwork, mechanical and electrical for the crushing plant, processing plant, conveying systems and infrastructure.

The layouts and model have been based on recently completed facility designs, modified construction and as-built drawings of past project facilities, as well as initial concept drawings and computer modelling.

Unit rates that reflect the current market conditions have been established for bulk materials, capital equipment and labour via an extensive Budget Quotation Request (BQR) process. Labour rates from the market have been benchmarked against in-house labour gang rates and indirect cost modelling to ensure adherence suitability to the current projects market. The rates used in the estimate have been reviewed and deemed to reflect the current market conditions. Budget pricing for equipment and infrastructure facilities was obtained from suitable suppliers and contractors.

21.2.5 Quantity Development

The Project works were quantified to represent the defined scope of work and to enable the application of rates to determine costs. Allowances for earthworks compaction, waste, rolling margin and the like are included in the build-up of unit costs.

Quantity information was derived from a combination of sources and categorised to reflect the maturity of design information as follows:

- Study Engineering: ncludes quantities derived from concept or preliminary engineering for the purpose of the study. Includes equipment lists and redline mark-ups of previous project drawings / data by engineering.
- Estimated: Includes quantities derived from sketches or redline mark-ups of previous project drawings / data by estimating or similar project go-bys.

- Factored: Quantities derived from percentages applied as a factor derived from previous projects.

The derivation of quantities is provided in Table 21.2.5, weighted by value of the direct permanent works (i.e. excluding temporary works, construction services, commissioning assistance, engineering costs, escalation and contingency).

Table 21.2.5 Derivation of Quantities

Classification	Quantity	Unit	Study Engineering %	Estimated %	Factored %
Concrete	22,137	m³	10	90	-
Structural Steel	1,899	t	1	99	-
Platework	436	t	4	96	-
Field Erected Tanks	1139	t	2	98	-
Mechanical Equipment	1	lot	95	5	-
Piping - Plant	1	lot	-	-	100
Piping - Overland	17	km	52	48	-
E & I Plant	1	lot	-	100	-

21.2.6 Pricing Basis

Estimate pricing was derived from a combination of the following sources:

- Budget Quotation: Budget pricing solicited specifically for the study or project estimate.
- Database: Historical database pricing that is less than six months old.
- Estimated: Historical database pricing older than six months, escalated to the current estimate base date.
- Factored: Factored from costs with a basis.

Table 21.2.6 summarises the source of pricing by major commodity, weighted by value of the direct permanent works (excluding temporary works, construction services, commissioning assistance, EPCM costs and contingency), including supply and installation.

Table 21.2.6 Sources of Pricing

Classification	Budget Quotation %	Database %	Estimated %	Factored %
Concrete	-	99	1	-
Structural Steel	99	-	1	-
Platework	98	-	2	-
Mechanical Equipment	67	26	2	5
Piping - Plant	-	-	-	100
Piping - Overland	-	95	-	5
E & I Equipment	-	98	2	-

Pricing has been categorised by the following cost elements, as applicable, for the development of each estimate item.

Bulk Materials

This component covers all other materials, normally purchased in bulk form, for installation on the project. Costs include the purchase price ex-works, any off-site fabrication, and transport to site (unless otherwise stated), and over-supply for anticipated wastage.

Plant Equipment

This component represents prefabricated, pre-assembled, off-the-shelf types of mechanical or electrical equipment item. Pricing is inclusive of all costs necessary to purchase the goods ex-works; generally excluding delivery to site (unless otherwise stated) but including operating and maintenance manuals. Vendor representation and commissioning spares have been allowed for separately in the estimate.

Installation

This component represents the cost to install the plant equipment and bulk materials on site or to perform site activities. Installation costs are further divided between direct labour, equipment and construction indirect costs.

The labour component reflects the cost of the direct workforce required to construct the Project scope. The labour cost is the product of the estimated work hours spent on site multiplied by the cost of labour, inclusive of overtime premiums, statutory overheads, and payroll burden.

The equipment component reflects the cost of the construction equipment and running costs required to construct the Project. The equipment cost also includes cranes, vehicles, small tools, consumables, and PPE.

Construction indirect costs encompass the remaining cost of installation and include items such as offsite management, onsite staff and supervision above trade level, crane drivers, equipment and

labour mobilisation and demobilisation, Rest and Recreation (R&R), meals and accommodation costs.

The labour gang rates, and equipment costs estimated for each major trade commodity are shown in Table 21.2.7

Table 21.2.7 Standard Direct Labour Gang Rates

Item	Direct Labour (US$)	Equipment (US$)	Total Hourly Rate (US$)
Concrete Installation	5.00	2.00	7.00
SMP	17.00	8.32	25.32
Field Erected Tanks	14.00	5.00	19.00
Building Installation	12.50	5.00	17.50
Electrical & Instrumentation	12.00	8.00	20.00

Generally, for bulk commodities, lower to mid range vendor and contractor rates were used in the compilation of the estimate.

Freight

The freight estimate was derived from the following:

- The assessment of freight tonnes or bulk volume and number of containers from the quantities derived in the capital cost estimate and vendor supplied shipping lists and advice.
- The most likely country of origin.
- The application of rates solicited from the market specifically for the project that includes pre-shipment, ocean freight, and post-shipment overland freight.
- By calculation as a percentage of supply costs for items where freight tonnes and bulk volumes are yet to be determined.
- By benchmarking against similar projects located in similar regions.

21.2.7 Field Indirects

Project construction offices and establishment, communications, computers, IT services, servers, telephones and fuel facilities are not included in the capital estimate. B2Gold are in the process of establishing their construction team at site and these costs are considered sunk costs and have not been included in the capital estimate.

Construction services such as power, water, fuel and consumables are included.

Construction indirect costs for all direct labour is included in the installation rates for all works in the capital estimate. This is inclusive of meals, PPE, travel and clothing. Earthworks rates are inclusive of fuel, maintenance and running costs of machinery.

Construction equipment and project cranes are included in the capital cost estimate.

21.2.8 Engineering Services

The estimate for engineering design services is based on an assessment of the work content using the drawing and equipment lists prepared for the study as a guide. Design manhours will be estimated on a deliverable basis and benchmarked against previous Lycopodium experience for similar sized projects. Duration based costs such as project management, construction and commissioning support will be estimated using the project implementation schedule and previous project experience as a guide.

21.2.9 Owner's Costs

B2Gold undertake plant construction on a self perform basis. The Owners project management, construction team and construction equipment is included as part of the construction indirect costs calculated for the project.

The Owner's construction team will interact closely with operations management personnel recruited during the construction phase of the Project.

In addition to the above, the following allowances have been made in the estimate:

- Pre-production costs.
- First fills (grinding media, lubricants, fuel, and reagents).
- Opening stocks.
- Plant mobile equipment (the majority of this will be handed over from the construction team's mobile fleet following plant handover).
- Project spares.
- Vendor representative and training costs for the process plant.
- Working capital.

21.2.10 Spares

Spares have been calculated as a percentage of Mechanical & Electrical Equipment supply.

21.2.11 Duties, taxes and Insurances

Government Duties and Taxes

The capital estimate generally excludes all government taxes and duties.

Project Insurances

Project insurances have been excluded from the estimate.

21.2.12 Contingency

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed project cost. The contingency does not cover scope changes, design growth, etc., or the listed qualifications and exclusions.

Contingency has been applied to the estimate on a line-by-line basis as a deterministic allowance by assessing the level of confidence in each of the defining inputs to the item cost, these being engineering, estimate basis and vendor or contractor information, and then applying an appropriate weighting to each of the three inputs. It should be noted that contingency is not a function of the specified estimate accuracy and should be measured against the project total that includes contingency. The resultant contingency for the project is 13%.

A summary of contingency percentage for the major horizontal packages is provided in Table 21.2.8.

Table 21.2.8 Contingency Percentage Summary

Facility	%
Architectural	10
Earthworks	13
Concrete	12
Electrical	15
Instrumentation	13
Platework / Tankage	12
Mechanical Equipment	12
Piping	21
Structural Steelwork	12
Owners Costs	12
Engineering	10
Consultants	15
General	13
Project Total	**13**

The project total contingency includes additional minor components not detailed above.

21.2.13 Deferred Capital

Deferred capital costs are covered in Section 21.3. Deferred capital costs are not included in the Capital Cost Estimate detailed in Table 21.2.1.

21.2.14 Escalation

There is no allowance for project escalation in the Capital Cost Estimate.

21.2.15 Qualifications and Assumptions

The capital estimate is qualified by the following assumptions:

- The base date for the bulk of pricing for the estimate is second quarter 2015 (2Q 2015).
- Prices of materials and equipment with an imported content have been converted to US$ at the rates of exchange stated previously in this document. All pricing received has been entered into the estimate utilising native currencies wherever possible.
- The bulk earthworks commodity rates that include imported material are based on the assumption that suitable construction / fill materials will be available from borrow pits within 2 km of the work fronts.
- Engineering quantities and rates for the TSF structure, access road and pipeline corridor, pit diversion channel, sediment control structures and water storage structures have been provided by Knight Piésold. Sub-consultant design costs related to these work scopes for project implementation have also been included.
- There is no allowance for unforeseen blasting in the bulk earthworks cost estimates, given the results of the site geotechnical investigation.
- The estimate allows for supply of structural steel and minor platework from South East Asia.
- It has been assumed mobile equipment purchased and used by the owner's construction team will be handed over to the owners operations team upon completion of construction. No allowance for additional mobile equipment has been made for operations.
- Allowance has been made to pile foundations of the grinding building and leach and CIP tanks areas based on preliminary geotechnical advice.

21.2.16 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Permits and licences.
- Project sunk costs.

- Land purchase and relocation costs.
- Government taxes and duties
- Exchange rate variations.

21.3 Deferred Capital

Project deferred capital costs are summarised in Table 21.3.1. Deferred capital includes payments for mine fleet and power generating equipment lease financed as part on pre-production equipment purchases, increases in the mining fleet and subsequent TSF stage raises over the life of mine, and mine closure and rehabilitation costs. These costs are not included in the pre-production capital cost summary covered in Section 21.2.

Table 21.3.1 Deferred Capital Summary

Main Area	US$
Lease Payments	75,400,000
Infrastructure	51,000,000
Mining	32,900,000
Closure and Rehabilitation	20,400,000
Total	**179,700,000**

The deferred capital schedule over the life of mine is estimated as shown in Table 21.3.2

Table 21.3.2 Deferred Capital Schedule

Year	Grand Total (US$)
2018	41,250,000
2019	31,500,000
2020	28,500,000
2021	21,950,000
2022	4,350,000
2023	5,850,000
2024	6,300,000
2025	6,300,000
2026	5,800,000
2027	5,800,000
2028	15,800,000
2029	4,100,000
2030	1,100,000
2031	1,100,000
Grand Total	**179,700,000**

21.4 Project Implementation

21.4.1 Implementation Strategy

The Project implementation plan has been developed utilising B2Gold's preferred self construct approach. Engineering consultants will provide design and procurement assistance services with B2Gold self performing construction according to the Project Schedule.

The mine planning study is based on an owner operator strategy, whereby B2Gold will purchase and operate the mine fleet and drill and blast services. B2Gold's mining group will also undertake all bulk earthwork activities for the Project.

The current mine plan anticipates mining commencing three months before first ore to the mill, with a total mine life of nine and a half years and with the mill operating on material recovered from stockpiles for a further three years after completion of mining.

The implementation strategy is structured into four broad stages:

- Detailed design of the process plant and infrastructure.
- Procurement.
- Construction.
- Commissioning and handover.

The estimated durations of key project activities are provided in Table 21.4.1.

The durations shown are preliminary and are based on receipt of the relevant authorisations to proceed.

Table 21.4.1 Key Project Milestones

Milestone Description	Planned Completion
Engineering Award Date	Week 0
Mill Contract Award	Week 1
Accommodation Camp Construction Commencement	Week 11
Concrete Construction Commencement	Week 28
Tank Construction Commencement	Week 36
SMP Construction Commencement	Week 50
E&I Construction Commencement	Week 77
Mills Mechanical Installation Commencement	Week 73
Power On	Week 105
First Gold Pour	Week 124

Several items and activities are potentially on or close to the schedule critical path for design and construction including procurement, manufacturing, transport and installation of long lead mechanical items for the process plant and the plant power station. Their criticality will be investigated further during early engineering.

Table of Contents

			Page
22.0	**ECONOMIC ANALYSIS**		**22.1**
	22.1	Introduction	22.1
	22.2	Summary	22.1
	22.3	Principal Assumptions	22.2
		22.3.1 Input Parameters and Project Schedule	22.2
		22.3.2 Mineral Reserves and Mining	22.3
		22.3.3 Production Schedule and Costs	22.3
		22.3.4 US$ Capital Cost	22.6
	22.4	US$ Financial Model Results	22.6
		22.4.2 US$ 1,250 Au Case	22.10
	22.5	Sensitivity Analysis	22.11

TABLES

Table 22.3.1	Financial Model Assumptions	22.2
Table 22.3.2	Doré Selling Costs	22.2
Table 22.3.3	Life of Mine Production Summary	22.5
Table 22.3.4	Sustaining Capital Summary (US$ Million)	22.6
Table 22.4.1	Overall Results of Financial Model US$	22.6
Table 22.4.2	Summary of Annual Cash Flow and Project Value	22.8
Table 22.4.3	Summary of Operating and Total Cash Costs US$	22.9
Table 22.4.4	Cash Flow and Project Value for US$ 1,250 Au Case	22.10
Table 22.5.1	Gold Price Sensitivity	22.12

FIGURES

Figure 22.5.1	Fekola Project Cash Flow Sensitivity	22.11

22.0 ECONOMIC ANALYSIS

22.1 Introduction

An economic analysis of the Project assesses the economic viability and performance of the Project under the assumptions and results of the OFS. The analysis was based on mine plans and production schedules as presented in Sections 15 and 16 (Mineral Reserves and Mining Methods) of this report, recoveries as presented in Sections 13 (Processing and Metallurgy) and 17 (Recovery Methods), and capital and operating costs as presented in Section 21 (Capital and Operating Costs). This analysis is project based and therefore assumes 100% ownership on a pre-tax basis. All site related costs, plus royalties, are included in the total cash costs as presented. For details on state interest refer to Section 4.

The above information was compiled into the Project financial model. The costs portion of the model includes all mining, processing, general and administration, and distributable (power and laboratory) costs with input from the mining and processing production schedules plus appropriate labour, consumable, operating, maintenance, and other costs. Revenue is also based on the production schedules plus the process recovery, gold price, selling costs, and royalty input. The results of the financial model include the cash flow, income, and cash cost tables, as well as various measures of project value including discounted cash flow and internal rate of return.

A sensitivity analysis was performed using gold price, processing cost, mining cost, fuel cost, capital cost, and labour cost as the variables. The sensitivity assumed variations of up to 20% for all variables. A summary of the results is included in this section.

22.2 Summary

The Project financial model was compiled using the inputs from this OFS as presented in prior sections. Key assumptions were the mining and processing production, project capital costs, and mining, processing, and general costs. The financial model is pre-tax and assumes 100% ownership.

Project gold production averages 276,000 ounces per year over the life of the project with an average of 350,000 ounces per year over the first seven years. The capital costs include US$ 395M in pre-production capital, plus US$ 180M in sustaining capital including finance payments, tailings facility expansions, and rehabilitation and closure costs. Mining costs average US$ 2.99/t over the life of the mine. Production cost is US$ 18.29/oz in design conditions, and averages US$ 18.30 during the steady state production in years 2018 through 2029. General and administration costs are US$ 14.9M per year during the full mining years, decreasing to US$ 11.1M when the mining department is no longer operational.

The results of the model show robust results including a life of mine cash flow of US$ 1.66B, IRR of 35%, and an NPV of US$ 1.01B at 5%. The total cash cost including 7.65% royalties is US$ 652/oz over the life of the mine and US$ 518 over the first seven years of production (2018 to 2024).

The project has low sensitivity to capital, fuel, processing, mining, and labour costs. The project is typically sensitive to gold prices, but maintains an IRR of 33% at a US$ 1,250/oz gold price and 19% at a US$ 1,000/oz gold price.

22.3 Principal Assumptions

22.3.1 Input Parameters and Project Schedule

The basic financial and economic assumptions are shown in Table 22.3.1 below. Based on the project implementation plan it is assumed that the procurement and construction phase will start in Q3 2015 and be completed in late Q4 2017. This construction period of 29 months is a reasonable assumption considering the logistics and infrastructure available.

Table 22.3.1 Financial Model Assumptions

Parameter	Unit	Value
Gold Price	US$/toz	1,300
Diesel Price	US$/L	0.924
HFO Price	US$/L	0.825
Exchange Rate	XOF/US$	606.06
Exchange Rate	EUR/US$	0.91

Doré bars will be transported from the Project site to the yet to be determined refiner by a combination of highly secured transport vehicles and/or aircraft. The costs associated with transporting, insuring, and refining doré are listing in Table 22.3.2 below, along with the royalties of 7.65%. These costs are typical of the industry and are based on B2Gold's experience in its current projects.

Table 22.3.2 Doré Selling Costs

Item	Value
Refinery Payable Metal Recoveries	
Gold – Refinery Payable Recovery	99.95%
Non-payable Base Metal, Slag in Dore	3.00%
Dore Transportation, Security, Insurance	
Basic Charge (US$ per Lot)	1,270.50
Weight Surcharge (US$ per kg)	12.57
Refinery Charges	
Refinery Treatment Charge (US$ per net weight troy ounces)	0.42
Assay charge (US$ per Lot)	75.00
Estimate no. of lots per year	26.04
Early Settlement Fee – Finance Charge	0.04%
Other Costs of Sale	
Gold Transfer Charge (US$ per ounce)	0.02
Silver Transfer Charge (US$ per ounce)	0.02
Total Royalties,% of the Net Production Value (Net Sales)	**7.65%**

22.3.2 Mineral Reserves and Mining

The life of mine (LoM) and annual production schedules are from the mineral reserves and base case mining schedule established in this report in Sections 15 and 16. This schedule mines 49.2 Mt of ore over a period of 9.5 years, including pre-stripping. Annual mined tonnage reaches a maximum of 32 Mt/y, totalling 219.3 million waste tonnes and 268.5 million total mined tonnes.

Due to the accelerated mining rate the medium and low grade ore is stockpiled when it is not needed for immediate crusher feed. This results in an average processed grade of 3.1 g/t over the first seven years of processing and an average LoM grade of 2.35 g/t.

Mining costs are based on the expected productivity, operating costs, and labour rates for an open pit gold mine in Mali. Productivity and operating costs are based on data from the equipment manufacturers with appropriate factors applied for local conditions and experience. Labour rates are fully loaded and are based on a survey of local mines including all relevant overtime, holiday, and other costs. Mine department management, technical services, and other general costs are included. See Section 21 for details of the operating mining costs.

22.3.3 Production Schedule and Costs

The operating costs as presented in Section 21 are based on the design throughput, grade, and ore / waste characteristics. These vary slightly over time and result in varying costs and gold production throughout the mine life. The overall production schedule is shown in Table 22.3.3 below and the resulting production costs (total and per ounce) are shown in the results of the financial model (Section 22.4).

Mining costs vary from a low of approximately US$ 2.40 / mined tonne when the pit is shallow and waste dumps are low to a high of approximately US$ 3.50 / mined tonne when the pit is fully developed and waste dumps are near the maximum heights. Mining costs are also dependent on diesel costs. A price of US$ 0.92/l was assumed based on recent costs. The sensitivity analysis (see Section 22.5) indicates that overall project sensitivity to fuel and mining costs is low and thus is not a project risk.

Mine production is similar to the optimized schedule reported in Section 16 and assumes that pre-stripping activities start late in the third quarter of 2017. The mining rate will ramp up to the full 32 Mtpa pace by the first quarter of 2018. The reserves will be mined out in 9.5 years, after which medium and low grade stockpiles will be rehandled to the processing plant.

Processing costs vary much less than mining costs, only due to varying distributed costs (power and laboratory). Under design conditions the average processing cost is US$ 18.29/rock tonne as described in Section 21. Note that processing costs are dependent on power costs (see below) which account for over 50% of the processing costs and over 25% of the total site costs (US$ 10.15 of US$ 38.58). As with mining costs, the sensitivity analysis (Section 22.5) indicates that overall project sensitivity to fuel and processing costs is low and thus is not a project risk.

Processing production assumes commissioning in Q4 2017 with December as the first production month. Process availability ramps up starting at 50% for December 2017, increasing to 70% for January 2018, 85% for February 2018, and 94% for the rest of the life of mine. Recovery ramps up

with a reduction of 3% for December 2017, 2% for January 2018, 1% for February 2018, and to the design recovery equation as presented in Section 13 (including the 1% scaling factor) for the rest of the life of mine.

The production schedule including mining and processing is shown in Table 22.3.3.

Site General costs include non-production related departments such as administration, accounting, human resources, environmental, security, safety, transportation, housing, etc. Also included are parasitic power plant losses, international flights for expatriates, busses for local labour, and other general costs. The costs decrease from US$ 14.9M during the full production years to US$ 11.1M when the mining department closes.

Power generation costs are closely tied to the cost of heavy fuel oil (HFO). For this study a conservative price of US$ 0.825/l for HFO 380 was assumed. Note that current prices at B2Gold's other mines are in the range of US$ 0.55 to US$ 0.70/l, depending on transportation costs, local price controls, and HFO specification. The resulting power cost is US$ 0.20/kWh including all consumables, operating, and maintenance costs.

Table 22.3.3 Life of Mine Production Summary

Description	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM or Average
Calendar Days / Year	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	**6,205**
Scheduled Processing Days	-	-	30	365	365	365	365	365	365	365	365	365	365	365	365	115	-	**4,525**
PROCESSING																		
Scheduled Processing Hours	-	-	720	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	2,760	-	**109,000**
Scheduled Downtime Hours	-	-	180	51	51	51	51	51	51	51	51	51	51	51	51	12	-	**804**
Unscheduled Downtime Hours	-	-	180	715	475	475	475	475	475	475	475	475	475	475	475	150	-	**6,270**
Available Operating Hours	-	-	360	7,990	8,230	8,230	8,230	8,230	8,230	8,230	8,230	8,230	8,230	8,230	8,230	2,600	-	**102,000**
Availability%	-	-	50.0%	91.3%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.1%	-	**93.5%**
Utilization%	-	-	66.7%	91.8%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.6%	-	**94.2%**
Grinding Feed Rate [tph @ 100% Avail.] - Calc. Weighted Avg	-	-	400	486	486	486	486	486	486	486	486	486	486	486	486	441	-	**304**
% Ore Tonnes Rehandled (%)	-	-	100%	66%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	**63%**
Ore Tonnes Processed per Day	-	-	4,800	10,640	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	9,960	-	**10,900**
Ore Tonnes Processed	-	-	144,000	3,880,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	1,150,000	-	**49,200,000**
Gold Grade Processed (g/t)	-	-	3.94	2.72	3.21	2.52	3.17	3.19	2.69	3.13	2.13	1.98	1.80	1.00	0.90	1.48	-	**2.35**
Recovery - Gold (%)	-	-	90.5%	92.8%	93.2%	92.8%	93.2%	93.2%	93.0%	93.2%	92.6%	92.5%	92.4%	91.3%	91.1%	92.2%	-	**92.8%**
HG Ore Processed:	-	-																
Ore Tonnes Processed	-	-	144,000	2,790,000	3,800,000	3,380,000	3,650,000	3,470,000	2,860,000	3,160,000	1,850,000	1,880,000	1,150,000	140,000	2,800	281,000	-	**28,500,000**
Gold Grade Processed (g/t)	-	-	3.94	3.30	3.31	2.75	3.36	3.49	3.26	3.63	3.15	2.81	3.25	3.25	3.25	3.25	-	**3.26**
Recovery - Gold (%)	-	-	90.5%	93.0%	93.2%	92.9%	93.2%	93.3%	93.2%	93.4%	93.1%	92.9%	93.2%	93.2%	93.2%	93.2%	-	**92.9%**
Grinding Feed Rate [tph @ 100% Availability]	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	
MG Fresh Ore Processed:																		
Ore Tonnes Processed	-	-	-	1,100,000	200,000	622,000	349,000	532,000	1,140,000	841,000	2,150,000	2,120,000	2,580,000	129,000	6,400	339	-	**11,800,000**
Gold Grade Processed (g/t)	-	-	-	1.24	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	-	**1.25**
Recovery - Gold (%)	-	-	0.0%	91.6%	91.6%	91.6%	91.6%	91.6%	91.6%	91.6%	91.6%	91.6%	91.6%	91.6%	91.6%	91.6%	-	**91.6%**
Grinding Feed Rate [tph @ 100% Availability]	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	
LG Ore Processed:																		
Ore Tonnes Processed	-	-	-	-	-	-	-	-	-	-	-	-	279,000	3,730,000	3,990,000	864,000	-	**8,860,000**
Gold Grade Processed (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	0.90	0.90	0.90	0.90	-	**0.90**
Recovery - Gold (%)	-	-	-	-	-	-	-	-	-	-	-	-	91.1%	91.1%	91.1%	91.1%	-	**91.1%**
Grinding Feed Rate [tph @ 100% Availability]	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	
Other Ore Processed:																		
Ore Tonnes Processed	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Gold Grade Processed (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Recovery - Gold (%)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Grinding Feed Rate [tph @ 100% Availability]	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	485.8	
Gold Produced (troy ounces)	**-**	**-**	**16,500**	**315,000**	**384,000**	**300,000**	**380,000**	**382,000**	**322,000**	**375,000**	**253,000**	**236,000**	**214,000**	**117,000**	**106,000**	**50,200**	**-**	**3,450,000**
Silver Produced (troy ounces)	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
SURFACE MINE PRODUCTION																		
Fekola Pit - Total																		
High Grade Ore Tonnes Mined	-	-	351,000	3,770,000	3,880,000	2,950,000	3,550,000	3,430,000	2,870,000	3,040,000	1,930,000	2,080,000	685,000	-	-	-	-	**28,500,000**
High Grade Gold Grade Mined (g/t)	-	-	3.94	3.26	3.31	2.51	3.50	3.52	3.21	3.72	3.00	2.70	3.81	-	-	-	-	**3.26**
Medium, Low Grade Ore Tonnes Mined	-	-	252,000	2,630,000	2,650,000	2,910,000	2,700,000	2,360,000	2,110,000	1,880,000	1,350,000	1,520,000	264,000	-	-	-	-	**20,600,000**
Med and Low Grade Gold Grade Mined (g/t)	-	-	1.06	1.08	1.11	1.10	1.10	1.10	1.10	1.10	1.08	1.12	1.17	-	-	-	-	**1.10**
Total Ore Tonnes Mined (no subgrade)	-	-	602,000	6,400,000	6,530,000	5,860,000	6,260,000	5,790,000	4,980,000	4,930,000	3,280,000	3,600,000	949,000	-	-	-	-	**49,200,000**
Total Gold Grade Mined (g/t)	-	-	2.74	2.37	2.42	1.81	2.47	2.53	2.31	2.72	2.21	2.03	3.08	-	-	-	-	**2.35**
Contained Gold Mined (g)	-	-	1,650,000	15,200,000	15,800,000	10,600,000	15,400,000	14,700,000	11,500,000	13,400,000	7,260,000	7,300,000	2,920,000	-	-	-	-	**116,000,000**
Contained Gold Mined (troy ounces) (31.1035)	-	-	53,000	487,000	507,000	340,000	496,000	471,000	371,000	430,000	233,000	235,000	93,900	-	-	-	-	**3,720,000**
Topsoil / Organic Layer Tonnes Mined	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Waste Tonnes Mined (FEKOLA)	-	-	6,430,000	25,600,000	25,500,000	26,100,000	25,700,000	26,100,000	26,800,000	27,000,000	20,800,000	8,460,000	943,000	-	-	-	-	**219,000,000**
Total Rock Tonnes Mined	-	-	7,030,000	32,000,000	32,100,000	31,900,000	32,000,000	31,900,000	31,800,000	31,900,000	24,000,000	12,100,000	1,890,000	-	-	-	-	**269,000,000**
Total Rock Tonnes Mined per Day	-	-	19,300	87,600	87,800	87,500	87,600	87,400	87,000	87,500	65,900	33,000	5,180	-	-	-	-	
Strip Ratio (Waste : Ore Tonnes)	-	-	10.7	4.0	3.9	4.5	4.1	4.5	5.4	5.5	6.3	2.4	1.0	-	-	-	-	**4.5**

22.3.4 US$ Capital Cost

Details of the preproduction capital costs are presented in section 21. In the financial model the costs are assumed to start in Q3 2015 and continue through November 2017.

Sustaining capital includes lease payments, additions to the mine fleet, tailings facility expansions, and rehabilitation / closure costs. Table 22.3.4 below summarizes the post-production capital costs.

Table 22.3.4 Sustaining Capital Summary (US$ Million)

	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	Total
Lease Payments	20.1	19.3	18.4	17.6	-	-	-	-	-	-	-	-	-		75.4
Mining Equipment	16.25	8.5	6.3	0.35	0.25	1.25	-	-	-	-	-	-	-		32.9
Tailings Storage Facility	4	3.5	3.7	3.8	3.9	4.5	5.1	5	5.5	5.5	6.5	-	-		51
Closure & Rehabilitation	0.9	0.2	0.1	0.2	0.2	0.1	1.2	1.3	0.3	0.3	9.3	4.1	1.1	1.1	20.4
Annual Total	**41.25**	**31.5**	**28.5**	**21.95**	**4.35**	**5.85**	**6.3**	**6.3**	**5.8**	**5.8**	**15.8**	**4.1**	**1.1**	**1.1**	**179.7**

22.4 US$ Financial Model Results

Table 22.4.1 below shows the overall outcomes of the Project OFS financial model. Note that this is a project based analysis and assumes 100% ownership on a pre-tax basis.

Table 22.4.1 Overall Results of Financial Model US$

Description	OFS - June 2015
SURFACE MINING	
Start of Mining (Prestrip)	Q3 2017
High Grade Ore Tonnes Mined grade	28,500,000
High Grade Gold Grade Mined (g/t)	3.26
Low and Medium Grade Ore Tonnes Mined	20,600,000
Low and Medium Grade Gold Grade Mined (g/t)	1.10
Total Ore Tonnes Mined	49,200,000
Total Gold Grade Mined (g/t)	2.35
Contained Gold Mined (troy ounces)	3,720,000
Waste Tonnes Mined - Total	219,000,000
Total Rock Tonnes Mined	269,000,000
Average Rock Tonnes Mined per Day	76,200
Sustained Rock Tonnes Mined per Day (2018-2024)	87,500
Average Rock Tonnes Mined per Year	27,800,000
Sustained Rock Tonnes Mined per Year	32,000,000
Strip Ratio (Waste : Ore Tonnes)	4.46

Description	OFS - June 2015
PROCESSING	
LOM Ore Tonnes Processed	49,200,000
LOM Gold Production (troy ounces)	3,450,000
1st 7 Years (2018-2024) Gold Production (average)	352,000
Maximum Annual Gold Production (troy ounces)	384,000
Ore Tonnes Processed per Year	4,000,000
Average Ore Tonnes Processed per Day	10,900
Average Gold Grade Processed (g/t)	2.35
Average Recovery - Gold (%) (Weighted Average)	92.8%
Average Availability% (including ramp-up)	93.5%
Grinding Feed Rate tph - Calc.Avg	484
% Ore Tonnes Rehandled (%)	63%
Process Plant Life (years)	12.3
ECONOMICS - PRE-TAX, 100% BASIS, CASH COSTS, NO LEASE OR FINANCE	
ASSUMPTIONS	
Gold Price (US$/oz)	1,300
Exchange Rate (CIF:US$)	606
Fuel Prices:	
• Diesel Fuel Cost (US$/L)	0.924
• HFO Fuel Cost (US$/L)	0.825
OPERATING CASH COST PER OUNCE	
Operating Cash Cost per Gold Ounce Produced	552
Total Cash Cost per Gold Ounce Produced	652
DISCOUNTED CASH FLOW (Q3 2015 - 2031)	
IRR	35%
NPV @ 0%	1,660,000,000
NPV @ 5%	1,010,000,000
NPV @ 10%	615,000,000
PAYBACK (approximate, at 0%)	2.2 years
Average Annual Revenue (2018 - 2029)	367,000,000
UNIT OPERATING COST PER TONNE	
Mining Operating Cost per Rock Tonne Mined	2.99
Mining Operating Cost per Tonne Processed	16.32
Processing Operating Cost per Tonne Processed	18.46
Administration Operating Cost per Tonne Processed	3.81
Total Operating Cost per Tonne Processed	**38.58**
SUSTAINING CAPITAL	
Lease Payments	75,400,000
Mining Equipment	32,900,000
Tailings Storage Facility	51,000,000
Closure and Rehabilitation	20,400,000
Total Sustaining Capital	**179,700,000**
ELECTRIC POWER	
kWh Generated	2,490,000,000
Type of Electric Power	Self Generation - HFO
Power Cost - US$ / kWh	0.20
Power Cost - US$ / tonne processed	10.15
Power Cost - US$ / Gold ounce Produced	144.68

The cash flow, Internal Rate of Return (IRR) and Net Present Value (NPV) were calculated from the resulting annual cash flows at discount rates of 0%, 5%, and 10%.

See Table 22.4.2 below for the results and details.

Table 22.4.2 Summary of Annual Cash Flow and Project Value

Description	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM
REVENUE																		
Gross Production Value	-	-	21,500,000	409,000,000	499,000,000	390,000,000	494,000,000	497,000,000	418,000,000	488,000,000	330,000,000	307,000,000	278,000,000	152,000,000	138,000,000	65,200,000	-	4,490,000,000
Less: Refining, Transportation Costs	-	-	-71,500	-729,000	-882,000	-697,000	-874,000	-878,000	-745,000	-862,000	-594,000	-555,000	-506,000	-293,000	-268,000	-146,000	-	-8,100,000
NET PRODUCTION VALUE	**-**	**-**	**21,400,000**	**408,000,000**	**499,000,000**	**390,000,000**	**494,000,000**	**496,000,000**	**418,000,000**	**487,000,000**	**329,000,000**	**306,000,000**	**277,000,000**	**152,000,000**	**137,000,000**	**65,100,000**	**-**	**4,480,000,000**
																	-	
COST OF PRODUCTION	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Surface Mining -	-	-	-36,600,000	-76,900,000	-78,900,000	-81,500,000	-85,300,000	-88,700,000	-92,600,000	-98,900,000	-85,400,000	-58,100,000	-19,600,000	-	-	-	-	-802,000,000
Surface Mining -	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing	-	-	-7,250,000	-71,900,000	-72,700,000	-72,900,000	-72,900,000	-73,200,000	-73,400,000	-73,300,000	-72,900,000	-72,900,000	-73,200,000	-73,400,000	-73,400,000	-24,100,000	-	-908,000,000
Site General	-	-	-10,100,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-13,700,000	-11,100,000	-11,100,000	-7,270,000	-	-187,000,000
Total Cost of Production	**-**	**-**	**-53,900,000**	**-164,000,000**	**-167,000,000**	**-169,000,000**	**-173,000,000**	**-177,000,000**	**-181,000,000**	**-187,000,000**	**-173,000,000**	**-146,000,000**	**-107,000,000**	**-84,600,000**	**-84,600,000**	**-31,400,000**	**-**	**-1,900,000,000**
																	-	
OTHER OPERATING COSTS	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Royalties	-	-	-1,640,000	-31,300,000	-38,200,000	-29,800,000	-37,800,000	-38,000,000	-32,000,000	-37,300,000	-25,200,000	-23,400,000	-21,200,000	-11,600,000	-10,500,000	-4,980,000	-	-343,000,000
Withholding Taxes	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-	-2,400,000
Other (Income)/Expense	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Other Operating Costs	**-150,000**	**-150,000**	**-1,790,000**	**-31,400,000**	**-38,300,000**	**-30,000,000**	**-38,000,000**	**-38,200,000**	**-32,100,000**	**-37,400,000**	**-25,300,000**	**-23,600,000**	**-21,400,000**	**-11,800,000**	**-10,700,000**	**-5,130,000**	**-**	**-345,000,000**
																	-	
PRE-PRODUCTION COSTS & OPERATING CAPEX	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
CapX: Complete Construction	-66,000,000	-202,000,000	-127,000,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-395,000,000
CapX: Sustaining Capital	-	-	-	-41,300,000	-31,400,000	-28,500,000	-22,000,000	-4,340,000	-5,890,000	-6,350,000	-6,240,000	-5,770,000	-5,840,000	-15,800,000	-4,110,000	-1,080,000	-1,100,000	-179,600,000
Purchase of Supplies Inventory	-	-	-11,000,000	-4,000,000	-	-	-	-	-	-	3,000,000	5,000,000	3,000,000	2,000,000	1,000,000	1,000,000	-	-
TOTAL CAPEX	**-66,000,000**	**-202,000,000**	**-138,000,000**	**-45,300,000**	**-31,400,000**	**-28,500,000**	**-22,000,000**	**-4,340,000**	**-5,890,000**	**-6,350,000**	**-3,240,000**	**-773,000**	**-2,840,000**	**-13,800,000**	**-3,110,000**	**-**	**-1,100,000**	**-575,000,000**
NET CASHFLOW ON SITE OPERATING ACTIVITIES	**-66,200,000**	**-202,000,000**	**-172,000,000**	**-168,000,000**	**262,000,000**	**162,000,000**	**261,000,000**	**277,000,000**	**199,000,000**	**256,000,000**	**127,000,000**	**136,000,000**	**147,000,000**	**41,500,000**	**39,000,000**	**28,400,000**	**-1,100,000**	**1,660,000,000**

SITE IRR AND NPV		
From Q3 2015 through 2030		35%
NPV - Net Present Value [% Discount] - US$	0%	1,660,000,000
NPV - Net Present Value [% Discount] – US$	5%	1,010,000,000
NPV - Net Present Value [% Discount] - US$	10%	615,000,000

The cash costs are calculated from the Fekola site costs including mining, processing, distributed, site general and administration, and other site related costs in addition to stockpile credits. See Table 22.4.3 below.

Table 22.4.3 Summary of Operating and Total Cash Costs US$

Description	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM
Surface Mining	-	-	36,600,000	76,900,000	78,900,000	81,500,000	85,300,000	88,700,000	92,600,000	98,900,000	85,400,000	58,100,000	19,600,000	-	-	-	-	802,000,000
Processing	-	-	7,250,000	71,900,000	72,700,000	72,900,000	72,900,000	73,200,000	73,400,000	73,300,000	72,900,000	72,900,000	73,200,000	73,400,000	73,400,000	24,100,000	-	908,000,000
Mine General	-	-	10,100,000	14,900,000	14,900,000	14,900,000	14,900,000	14,900,000	14,900,000	14,900,000	14,900,000	14,900,000	13,700,000	11,100,000	11,100,000	7,270,000	-	187,000,000
Change in Stockpiled Ore	-	-	-27,800,000	-17,600,000	-26,600,000	-27,400,000	-31,000,000	-31,800,000	-30,400,000	-31,500,000	-10,700,000	14,300,000	53,500,000	73,100,000	73,100,000	20,900,000	-	-
Refinery Charge	-	-	31,700	569,000	694,000	543,000	687,000	691,000	582,000	678,000	459,000	427,000	387,000	213,000	193,000	92,400	-	6,250,000
Dore Transportation, Security, Insurance	-	-	39,700	160,000	188,000	154,000	186,000	187,000	163,000	184,000	135,000	128,000	119,000	80,100	75,700	53,300	-	1,850,000
Supplies Inventory Exchange Rate Adjustment	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
B2Gold Corporate Adjustment	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL OPERATING CASH COST	-	-	**26,100,000**	**147,000,000**	**141,000,000**	**143,000,000**	**143,000,000**	**146,000,000**	**151,000,000**	**156,000,000**	**163,000,000**	**161,000,000**	**161,000,000**	**158,000,000**	**158,000,000**	**52,500,000**	-	**1,910,000,000**
	-	-															-	
Minerals Act - Royalty	-	-	1,640,000	31,300,000	38,200,000	29,900,000	37,800,000	38,000,000	32,000,000	37,300,000	25,200,000	23,500,000	21,300,000	11,600,000	10,500,000	4,980,000	-	343,000,000
TOTAL OTHER CASH COST	-	-	**1,640,000**	**31,300,000**	**38,200,000**	**29,900,000**	**37,800,000**	**38,000,000**	**32,000,000**	**37,300,000**	**25,200,000**	**23,500,000**	**21,300,000**	**11,600,000**	**10,500,000**	**4,980,000**	-	**343,000,000**
GOLD PRODUCED (OUNCES)	-	-	**16,500**	**315,000**	**384,000**	**300,000**	**380,000**	**382,000**	**322,000**	**375,000**	**253,000**	**236,000**	**214,000**	**117,000**	**106,000**	**50,200**	-	**3,450,000**
	-	-															-	
Operating Cash Cost per Gold ounce Produced	-	-	**1,600**	**467**	**367**	**475**	**376**	**382**	**470**	**417**	**643**	**682**	**751**	**1,400**	**1,500**	**1,000**	-	**552**
Total Cash Cost per Gold ounce Produced	-	-	**1,700**	**566**	**466**	**574**	**475**	**481**	**569**	**516**	**742**	**781**	**851**	**1,500**	**1,600**	**1,100**	-	**652**

22.4.2 US$ 1,250 Au Case

Due to the current gold price the cash flow, IRR, and NPV results of the financial model assuming US$ 1,250/oz gold price are shown below.

Table 22.4.4 Cash Flow and Project Value for US$ 1,250 Au Case

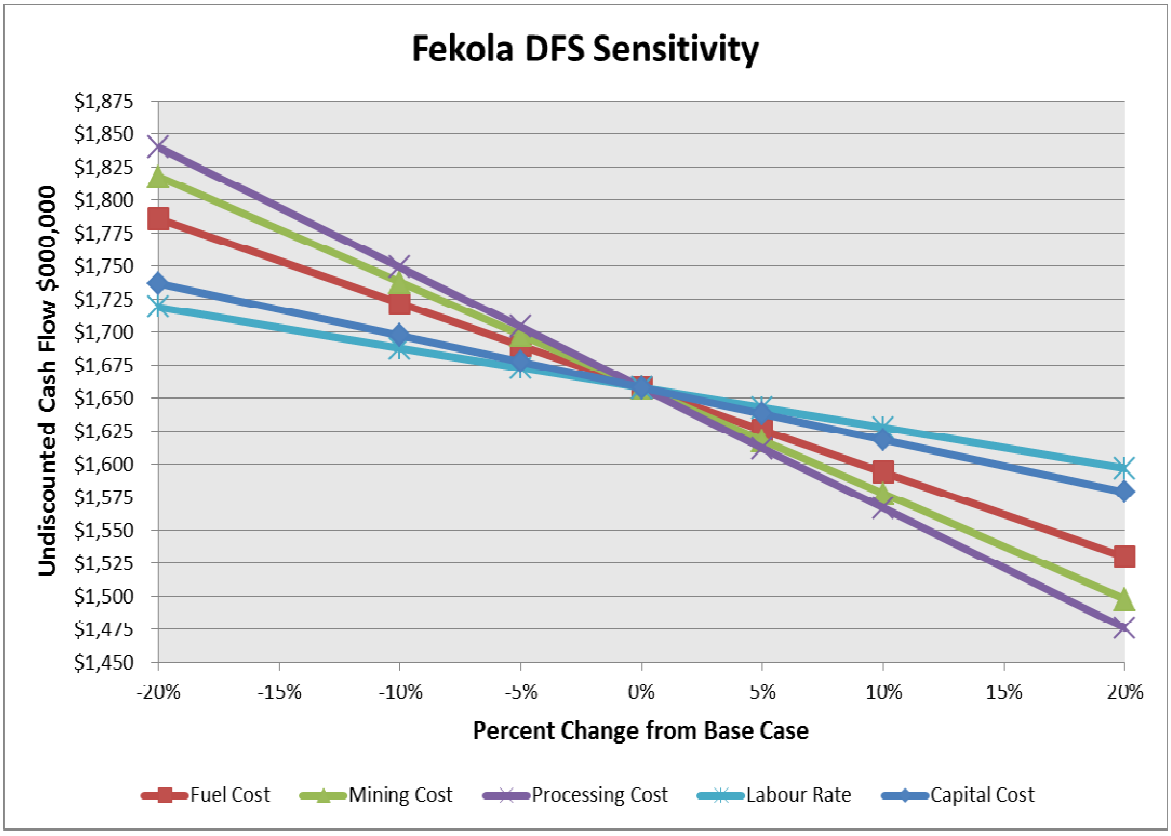
Description	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM
REVENUE																		
Gross Production Value	0	0	20,600,000	393,000,000	480,000,000	375,000,000	475,000,000	478,000,000	402,000,000	469,000,000	317,000,000	295,000,000	267,000,000	146,000,000	132,000,000	63,000,000	0	4,310,000,000
Less: Refining, Transportation Costs	0	0	-70,700	-714,000	-865,000	-683,000	-857,000	-861,000	-730,000	-845,000	-583,000	-544,000	-497,000	-287,000	-264,000	-143,000	0	-7,940,000
NET PRODUCTION VALUE	**0**	**0**	**20,600,000**	**392,000,000**	**479,000,000**	**375,000,000**	**475,000,000**	**477,000,000**	**402,000,000**	**468,000,000**	**316,000,000**	**294,000,000**	**267,000,000**	**146,000,000**	**132,000,000**	**62,600,000**	**0**	**4,310,000,000**
COST OF PRODUCTION																		
Surface Mining -	0	0	-36,600,000	-76,900,000	-78,900,000	-81,500,000	-85,300,000	-88,700,000	-92,600,000	-98,900,000	-85,400,000	-58,100,000	-19,600,000	0	0	0	0	-802,000,000
Processing	0	0	-7,250,000	-71,900,000	-72,700,000	-72,900,000	-72,900,000	-73,200,000	-73,400,000	-73,300,000	-72,900,000	-72,900,000	-73,200,000	-73,400,000	-73,400,000	-24,100,000	0	-908,000,000
Site General	0	0	-10,100,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-14,900,000	-13,700,000	-11,100,000	-11,130,000	-7,300,000	0	-187,000,000
Total Cost of Production	**0**	**0**	**-53,900,000**	**-164,000,000**	**-167,000,000**	**-169,000,000**	**-173,000,000**	**-177,000,000**	**-181,000,000**	**-187,000,000**	**-173,000,000**	**-146,000,000**	**-107,000,000**	**-84,600,000**	**-84,600,000**	**-31,400,000**	**0**	**-1,900,000,000**
OTHER OPERATING COSTS																		
Royalties	0	0	-1,580,000	-30,100,000	-36,700,000	-28,700,000	-36,400,000	-36,500,000	-30,800,000	-35,900,000	-24,200,000	-22,500,000	-20,400,000	-11,200,000	-10,100,000	-4,790,000	0	-330,000,000
Withholding Taxes	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	0	-2,400,000
Other (Income)/Expense	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Other Operating Costs	**-150,000**	**-150,000**	**-1,730,000**	**-30,200,000**	**-36,900,000**	**-28,800,000**	**-36,500,000**	**-36,700,000**	**-30,900,000**	**-36,000,000**	**-24,400,000**	**-22,700,000**	**-20,600,000**	**-11,300,000**	**-10,300,000**	**-4,940,000**	**0**	**-332,000,000**
PRE-PRODUCTION COSTS & OPERATING CAPEX																		
CapX: Complete Construction	-66,000,000	-202,000,000	-127,000,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	-395,000,000
CapX: Sustaining Capital	0	0	0	-41,300,000	-31,400,000	-28,500,000	-22,000,000	-4,340,000	-5,890,000	-6,350,000	-6,240,000	-5,770,000	-5,840,000	-15,800,000	-4,110,000	-1,080,000	-1,100,000	-180,000,000
Purchase of Supplies Inventory	0	0	-11,000,000	-4,000,000	0	0	0	0	0	0	3,000,000	5,000,000	3,000,000	2,000,000	1,000,000	1,000,000	0	0
TOTAL CAPEX	**-66,000,000**	**-202,000,000**	**-138,000,000**	**-45,300,000**	**-31,400,000**	**-28,500,000**	**-22,000,000**	**-4,340,000**	**-5,890,000**	**-6,350,000**	**-3,240,000**	**-773,000**	**-2,840,000**	**-13,800,000**	**-3,110,000**	**-76,500**	**-1,100,000**	**-575,000,000**
NET CASHFLOW ON SITE OPERATING ACTIVITIES	**-66,200,000**	**-202,000,000**	**-173,000,000**	**153,000,000**	**245,000,000**	**148,000,000**	**243,000,000**	**259,000,000**	**184,000,000**	**239,000,000**	**116,000,000**	**125,000,000**	**137,000,000**	**36,100,000**	**34,100,000**	**26,100,000**	**-1,100,000**	**1,500,000,000**

SITE IRR AND NPV		
From Q3 2015 through 2030		33%
NPV - Net Present Value [% Discount] – US$	0%	1,500,000,000
NPV - Net Present Value [% Discount] – US$	5%	901,000,000
NPV - Net Present Value [% Discount] - US$	10%	542,000,000

22.5 Sensitivity Analysis

The processing, mining, fuel, capital, and labour costs as well as the gold price were varied by 20% (positive and negative) to determine the sensitivity of the project. See Figure 22.5.1 below. The project is least sensitive to labour and capital costs, with increases in costs of 20% resulting in decreases in project cash flow being reduced by approximately 5%. Sensitivity to increases in fuel, mining, and processing costs is somewhat higher with 20% changes resulting in decreased cash flow of up to 11%. None of these would be considered sensitive.

Figure 22.5.1 Fekola Project Cash Flow Sensitivity



The project is somewhat sensitive to the gold price, but maintains IRR of 19% even with gold prices at US$ 1,000/oz. See Table 22.5.1 below.

Table 22.5.1 Gold Price Sensitivity

Gold Price US$ /oz	Pre-Tax NPV ($000,000)			IRR %
	0%	5%	10%	
900	387	162	30	12%
1,000	706	373	176	19%
1,100	1,024	584	322	25%
1,200	1,342	796	469	30%
1,250	1,502	901	542	33%
1,300	**1,661**	**1,007**	**615**	**35%**
1,400	1,979	1,218	761	40%
1,500	2,298	1,430	907	44%
1,800	3,253	2,064	1,345	56%

Table of Contents

Page

23.0 ADJACENT PROPERTIES **23.1**
23.1 Regional Gold Deposits 23.1

FIGURES
Figure 23.1.1 Regional Map of Known Gold Deposits 23.1

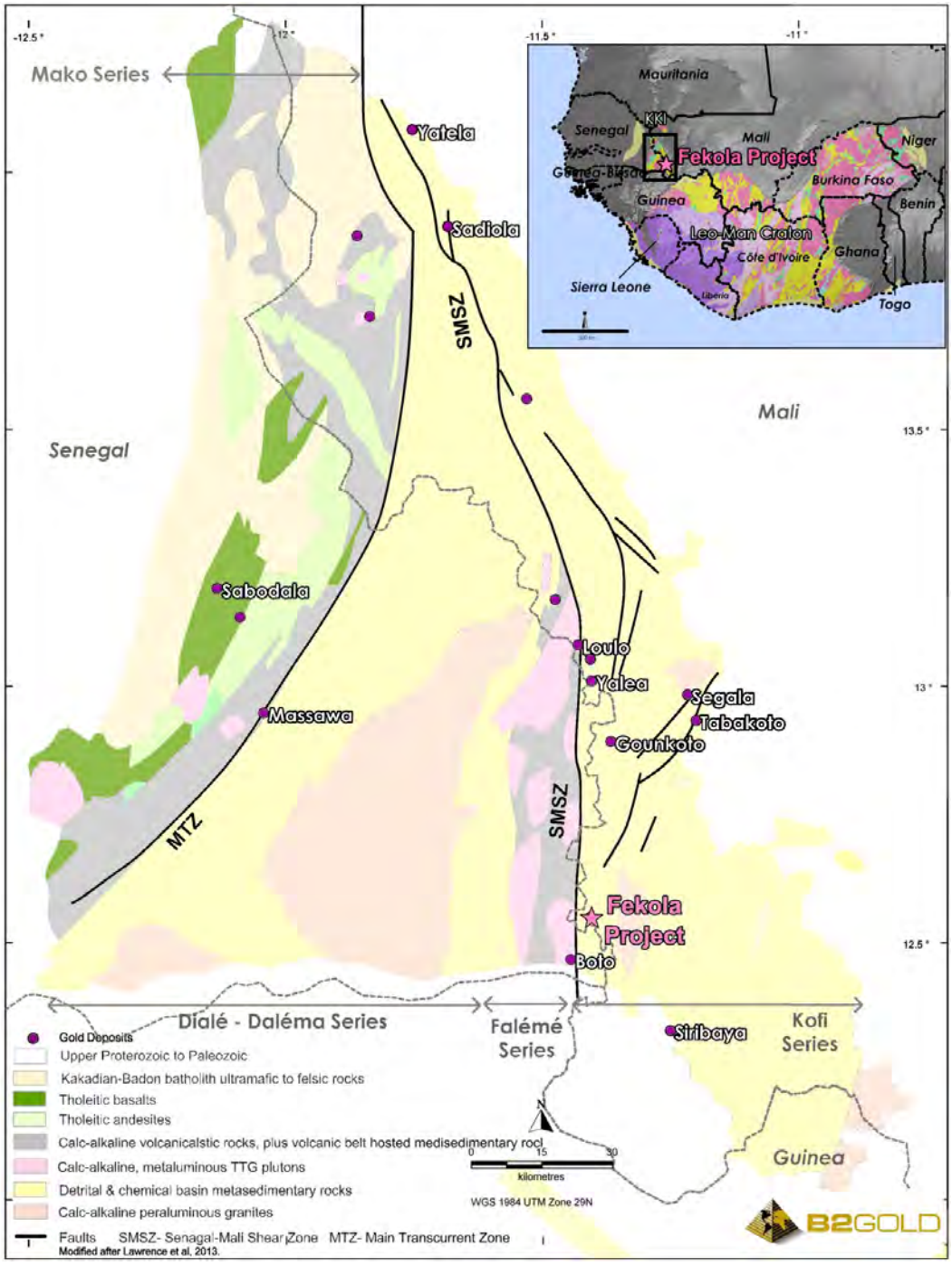
23.0 ADJACENT PROPERTIES

23.1 Regional Gold Deposits

There are no properties adjacent to the Fekola deposits that are relevant to the project.

Figure 23.1.1 shows the location of known gold deposits in the region.

Figure 23.1.1 Regional Map of Known Gold Deposits



FEKOLA GOLD PROJECT

NI 43-101 - OTHER RELEVANT DATA AND INFORMATION

3174-STY-002

Table of Contents

			Page
24.0	**OTHER RELEVANT DATA AND INFORMATION**		**24.1**
	24.1	Other Relevant Data	24.1
	24.2	Risks and Opportunities	24.1
		24.2.1 Identified Opportunities	24.1

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Other Relevant Data

The cyanide destruction testwork report received late in the study phase recommended the cyanide destruction circuit design include two reactor tanks operating in parallel. Plant capital and operating costs have been updated to reflect the change to two tanks, however the study design deliverables (flowsheets, equipment lists and layout drawings) only represent a single reactor design. The cyanide destruction circuit will be updated to correctly reflect testwork recommendations as part of the project detailed design.

24.2 Risks and Opportunities

24.2.1 Identified Opportunities

A mill vendor has advised of availability of an unused SAG mill that is larger (36 x 20 ft) than the mill size recommended at part of the original study comminution modelling (32 x 16 ft). Additional comminution modelling completed with the larger SAG mill in circuit has concluded this option to be favourable, due to the following opportunities:

- Minimal increase in capital cost (estimate at US$ 2.8M).
- Decrease in mill media consumption and therefore plant operating consumable costs (estimated at US$ 1.5M/year). Due to operation with a lower ball charge.
- Minimal increase in power consumption in the overall grinding circuit (estimate as 0.1 kWh/t milled).
- Provides installed capacity within the grinding circuit to allow plant throughput expansion in the future.

FEKOLA GOLD PROJECT

NI 43-101 - CONCLUSIONS

3174-STY-002

Table of Contents

			Page
25.0	**CONCLUSIONS**		**25.1**
	25.1	Conclusions	25.1

25.0 CONCLUSIONS

25.1 Conclusions

The OFS has identified, defined and costed a low technical risk conventional open cut mine and carbon in pulp processing facility.

Commercial gold mining has been established in West Africa for over 100 years, and Mali and surrounding countries have varying levels of established mining infrastructure.

Sovereign risk is understood and mitigated by the relatively short payback of the Project. Residual environmental and social issues are low and manageable.

The Project is economically viable at current gold prices. Applying a long term gold price of US$ 1,300/oz on a flat line basis from the commencement of production, the project has an estimate Net Present Value (NPV) pre-tax of US$1.01B at a 5% discount rate, generating a pre-tax internal rate of return (IRR) of 35% and a project payback of approximately 2.25 years from first gold production. The initial mine life is estimated at 12.5 years.

The February 2015 Mineral Resource estimate forms the basis for the OFS and incorporates drilling results available at January 24, 2015. Probable mineral reserves of 49.17 Mt at an average grade of 2.35 g/t Au containing 3.72 Moz of gold have been established.

The OFS estimates annual gold production averaged over the first seven years to be 350,000 oz of gold per year at an average operating cash cost of US$ 418/oz. The averaged LOM estimate for annual gold production is 276,000 oz of gold per year at an average operating cash cost of US$ 552/oz. Total pre-production capital costs are estimated at US$ 395M plus an additional US$ 67M for fleet and generator costs which are expected to be leased financed.

The processing flowsheet has been selected based on metallurgical test work results and consists of single stage primary crushing, a SABC grinding circuit (SAG and Ball mill with recycle pebble crushing), leach feed thickening with thickener overflow treated through a carbon in column (CIC) circuit, leaching followed by carbon in pulp (CIP) adsorption, elution and gold recovery to doré, cyanide destruction, tailings thickening and disposal circuits.

The plant facility and support infrastructure is designed to process 4 Mtpa of ore, and incorporates a 25% design margin to allow for future increases in circuit operation with minimum additional capital expenditure. Selected mill sizing also incorporates additional grinding capacity and can support plant throughput expansion beyond 5 Mtpa should exploration potential and future economic evaluation return a positive case for further expansion.

A project sensitivity analysis was performed on gold price, fuel price, labour costs, processing costs and mining costs. The Project shows low sensitivities to all inputs except for the gold price. Note that even with a low gold price of US$ 1,000/oz the Project maintains a pre-tax IRR of 19%.

Based on the Project's robust economics, B2Gold have already demonstrated a desire to move forward with project execution through the approval and commencement of project early works. The scope currently underway includes:

- Site construction early works, encompassing:
  - construction of the new mine access from the existing Millennium Highway to site. Road construction is well underway with completion expected prior to the 2015 wet season to enable continued all weather access to site
  - construction of the airstrip
  - clearing and grubbing of areas for plant and camp facilities
  - mobilisation of construction offices/facilities and equipment.
- Front End Engineering Design, including:
  - identification and tendering of key long lead equipment packages, such that orders can be placed upon final project execution approval. Key packages include; mills, crushers, thickeners, tanks, power station and the accommodation camp
  - development of project piping and instrumentation drawings (P&IDs) and establishment of engineering design basis and specification documents.

FEKOLA GOLD PROJECT

NI 43-101 - RECOMMENDATIONS

3174-STY-002

Table of Contents

			Page
26.0	**RECOMMENDATIONS**		**26.1**
	26.1	Recommendations	26.1

TABLES

Table 26.1.1	Key Project Milestones	26.1

26.0 RECOMMENDATIONS

26.1 Recommendations

Based on the conclusions of Section 25, it is recommended that B2Gold Corporation advance implementation of the Project as soon as possible.

The proposed project implementation approach of engineering consultants providing design services and B2Gold self performing construction has proven successful with previous B2Gold projects.

Site development will take place year round, utilising a work force of experienced Malian nationals, trained and supervised by African national and expatriate supervisors. Many of the African national and expatriate supervisors have worked on previous successful projects associated with B2Gold in Africa and will be familiar with the B2Gold construction approach as well as African working conditions.

Based on this implementation strategy and an end of June 2015 project approval, first gold pour can be achieved in December 2017. Key project milestones are summarised in Table 26.1.1, below.

Table 26.1.1 Key Project Milestones

Milestone Description	Planned Completion
Project Approval	Week 0
Engineering Award	Week 0
Mill Contract Award	Week 1
Accommodation Camp Construction Commencement	Week 11
Concrete Construction Commencement	Week 28
Tank Construction Commencement	Week 36
SMP Construction Commencement	Week 50
E&I Construction Commencement	Week 77
Mills Mechanical Installation Commencement	Week 73
Power On	Week 105
First Gold Pour	Week 124

Sufficient metallurgical testing and trade-off studies have been completed to confidently establish the recovery process and set the design criteria. Capital and operating costs have been estimated to a level of accuracy consistent with the OFS level and no significant environmental, social or permitting issues exist, that are known of, that would prevent exploitation of the Project deposits other than as discussed within this Report.

FEKOLA GOLD PROJECT

NI 43-101 - SELECTED REFERENCES

3174-STY-002

Table of Contents

			Page
27.0	**SELECTED REFERENCES**		**27.1**
	27.1	References	27.1

27.0 SELECTED REFERENCES

27.1 References

Lycopodium Minerals, Fekola Gold Project Definitive Feasibility Study, 3174-STY-001_A, compiled by Lycopodium Minerals Pty Ltd for B2Gold Corporation, June 2015.

Orway Mineral Consultants, Report 7602 Rev 0, Fekola Gold Project, SAB / SABC Options for 4 to 5 Mtpa, February 2015.

Orway Mineral Consultants, Report 7635 FINAL, Fekola Gold Project, SAG Mill Assessment, May 2015.

Knight Piésold, Report PE401-00079/04 Rev 0, Fekola Gold Project, Definitive Feasibility Study Report, compiled by Knight Piésold for B2Gold Corporation, June 2015.

Knight Piésold, Report PE401-00079_02 Rev 0, Feasibility Study Geotechnical Investigation Report, May 2015.

Knight Piésold, Memorandum PE15-00487, Fekola Gold Project – Water Balance Modelling, May 2015.

Mining Solutions Consultancy, Fekola Gold Project Definitive Feasibility Study, Open Pit Mining Section, compiled by Mining Solutions Consultancy Pty Ltd for B2Gold Corporation, May 2015.

SGS Mineral Services, Project CALR-14088-005 – Final Report 1 (Grindability), A Preliminary Grinding Circuit Design for the Fekola Project based on the Grindability Characteristics of 21 Samples, March 2015.

SGS Mineral Services, Project 14088-005 – Final Report, The Mineralogical Characteristics and Gold Deportment of Three Fekola Composite Samples, April 2015.

SGS Mineral Services, Project 14088-005 – Final Report – Revision 1, The Recovery of Gold from Fekola Project Samples, May 2015.

BBA, Technical Report, Document 3294004-000000-49-ERA-0001 / R00, Optimization of Gold Leaching from a Composite Sample from the Fekola Project, Mali, March 2015.

Jenike & Johanson, Report 70033-1, Fekola Project, Mali, Flow Property Test Results for Gold Ore, October 2014.

Jenike & Johanson, Report 70033-2, Fekola Project, Mali, Functional Recommendations for Stockpile Reclaim, April 2015.

FLSmidth, Laboratory Testing Report – Sedimentation and Rheology Tests for B2Gold – Fekola Gold Project on Leach and CND Tailings Thickener Duty, May 2015.

FLSmidth, Report KRTS20911, Gravity Test Work Report – B2Gold Fekola, December 2014.

FLSmidth, Gravity Circuit Modelling Report, B2Gold Corp., Fekola Project, January 2015.

Erik Devuyst Consultant Inc., Fekola Project - Cyanide Destruction Testwork and Basic Design Criteria, May 2015.

JEStanley, B2Gold Fekola Laboratory Mail, April 2015.

Standards Australia, 2004, Risk Management AS/NZS 4360:2004.

Standards Australia, 2004, Risk Management Guidelines – Companion to AS/NZS 4360:2004.

AusIMM, December 2012, JORC Code 2012 Edition.

Epoch Resources (Pty) Ltd, April 2013, Environmental and Social Impact Study (ESIS) for the Fekola Gold Project.

MPR Geological Consultants Pty Ltd, August 2013, Fekola Resource Estimate.

George, Orr and Associates, June 2104 (Appendix M1), Fekola Gold Project, Geotechnical Evaluation for Open Pit Mining Feasibility Purposes.

Mining Solutions Consultancy Pty Ltd, August 2014, Fekola Gold Project DFS Mining Study Report – DRAFT.

Knight Piésold, September 2014, Fekola Gold Project Bankable Feasibility Study Pit Dewatering Assessment.

B2Gold Corp, April 2015, Fekola Gold Project Mineral Resources Estimate.

B2Gold Corporation, 2014: Fekola Gold Project, Mali, NI 43-101 Technical Report on Preliminary Economic Assessment, 11 August 2014, 288 p.

B2Gold, A.Brown, Internal Memorandum, 2014 Fekola Metallurgical Sampling Program, November 2014.

B2Gold, D.Olsen, Internal Presentation, Fekola Met Review Notes, May 2015.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2003: Estimation of Mineral Resources and Mineral Reserves, Best Practice Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 23, 2003, http://www.cim.org/committees/estimation2003.pdf.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2010: CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 2010.

Canadian Securities Administrators (CSA), 2011: National Instrument 43-101, Standards of Disclosure for Mineral Projects, Canadian Securities Administrators.

MPR Geological Consultants Pty Ltd., 2013: Fekola Resource Estimate, Mali West Gold Project, Mali: report for Papillon Resources, February 2013, 73p.

MPR Geological Consultants Pty Ltd., 2013: Fekola Resource Estimate, Mali West Gold Project, Mali: report prepared for Papillon Resources, August 2013, 76 p.

Bassot, J.P., 1987. Le complexe volcano-plutonique calcoalcalin de la riviere Dalema (Est-Senegal): discussion de sa signification geodynamique dans le cadre de l'orogenie eburneene (Proterozoic inferieur). Journal of African Earth Science, Vol 6, pgs 353 to 367.

Boyd, A., Dahl, R., and Dowling, S., 2014: The Fekola Gold Deposit: A new multi-million ounce gold discovery in the Kéniéba District of Western Mali. New Gen Gold Conference Proceedings (2014): Case Histories of Discovery.

Dia. A., Van Schmus, W.R., and Kroner, A., 1997: Isotopic Constraints on the Age and Formation of a Paleo-Proterozoic Volcanic Arc Complex in the Kedougou Inlier, Eastern Senegal, West Africa: Journal of African Earth Sciences, vol 24, pgs 197 to 213; cited in Lawrence et al., 2013a.

Dorling, S., 2013: Geological Review, Fekola Au Deposit, Médinandi Project, W-Mali, Stratigraphic–Structural Assessment: report prepared by CSA Global Pty Ltd. for Papillon Resources, 28 March, 2013, 40 p.

Groves, D.I., Goldfarb, R.J., Gebre-Mariam, M., Hagemann, S.G., and Robert, F., 1999. Orogenic gold deposits: A proposed classification in the context of their crustal distribution and relationship to other gold deposit types. Ore Geology Reviews 13 (1998) pgs 7 to 27.

Groves, D.I., Goldfarb, R.J., Knox-Robinson, C.M., Ojala, J., Gardoll, S., Yun, G.Y., and Holyland, P., 2000. Late-kinematic timing of orogenic gold deposits and significance for computer-based exploration techniques with emphasis on the Yilgarn Block, Western Australia. Ore Geology Reviews 17 (2000), pgs 1 to 38.

Gueye, M., Ngom, P.M., Diene, M., Thiam, Y., Siegesmund, S., Wemmer, K., and Pawlig, S., 2008: Intrusive Rocks and Tectno-metamorphic Evolution of the Mako Paleoproterozoic Belt (Eastern Senegal, West Africa): Journal of African Earth Sciences, vol. 50, pgs 88 to 110

Hirdes W., and Davis, D.W., 2002: U-Pb Geochemistry of Paleoproterozoic Rocks in the Southern Part of the Kedougon-Kéniéba Inlier, Senegal, West Africa: Evidence of Diachronous Accretionary Development of the Eburnean Province: Precambrian Research, vol 118, pgs 83 to 89

Lawrence D., Treloar P, Rankin A., Harbidge P., and Holliday J., 2013: The Geology and Mineralogy of the Loulo Mining District, Mali, West Africa: Evidence for Two Distinct Styles of Orogenic Gold Mineralization: Economic Geology, v 108 pgs 199 to 227.

Ross, K., 2014. Petrographpic report on the Fekola gold deposit, Mali. Internal report prepared for B2Gold Corp.

Villeneuve M., and Cornée, J.J., 1994: Structure, Evolution and Paleoceanography of the West African Craton and Bordering Belts during the Neoproterozoic: Precambrian Research, v 69, pgs 307 to 326

Weinberg, R.F., Hodkiewicz, P.F. and Groves, D.I. 2004. What controls gold distribution in Archean terranes?. Geology v 32, Issue No. 7, pgs 545 to 548.

FEKOLA GOLD PROJECT

NI 43-101 - QP CERTIFICATES

3174-STY-002

Table of Contents

		Page
28.0	**QP CERTIFICATES**	**28.1**

28.0 QP CERTIFICATES

CERTIFICATE OF AUTHOR

I, Tom Garagan, P. Geo., do hereby certify that:

1. I am currently employed as a Senior Vice President of Exploration by B2Gold Corp ("**B2Gold**") with an office at Suite 3100, 595 Burrard Street, PO Box 49143, Vancouver, BC V7X 1J1.

2. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia, and of the Association of Professional Engineers, Geologists and Geophysicists of Alberta. I graduated from the University of Ottawa with a Bachelor of Science (Honours) degree in Geological Sciences in 1980. I have practiced my profession for 35 years. In this time I have been directly involved in generating and managing exploration activities, and in the collection, supervision and review of geological, mineralization, exploration and drilling data; geological models; sampling, sample preparation, assaying and other resource-estimation related analyses; assessment of quality assurance-quality control data and databases; and supervision of mineral resource estimates.

3. 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.

4. I am responsible for the preparation of the technical report entitled "B2Gold Corp NI 43-101 Technical Report Feasibility Study on the Fekola Gold Project in Mali" dated July 24, 2015, with an effective date of June 30, 2015, (the "**Technical Report**"), with specific responsibility for Sections 6 through 12, Section 14, and Section 23. I most recently visited the property on November 25, 2014 for 3 days.

5. My previous involvement with the property that is the subject of this Technical Report consists of acting as a Qualified Person for the technical report on the Preliminary Economic Assessment issued for the Fekola Gold Project, filed August 2014.

6. I am not independent of the issuer.

7. I have read National Instrument 43-101 and Form 43-101, and the Sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible

contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 24th day of July, 2015

(Signed) "*Tom Garagan*"

_____

Tom Garagan, P. Geo.

CERTIFICATE OF AUTHOR

I, Sandra (Sandy) Hunter, do hereby certify that:

1. I am currently employed as Principal Process Engineer by Lycopodium Minerals Pty Ltd with an office at 163 Leichhardt Street, Spring Hill, Queensland, 4000 Australia.

2. I am a Member of the Australasian Institute of Mining and Metallurgy (AusiMM), membership number 112041 and registered as a Chartered Professional with that Institute. I am a graduate of Murdoch University in 2000 with a Bachelor of Science degree in Mineral Science (Extractive Metallurgy) with Honours Class 2A. I have worked as a metallurgist, process plant manager and principal process engineer for a total of nineteen years. My relevant experience for the purpose of the Technical Report is review and report as a consultant on numerous process facilities and mining projects around the world. My experience includes working as a process engineer on a number of feasibility studies and projects, including commissioning, for both greenfield and expansion projects, for the mining industry in Africa, Australia and Asia. My operational experience includes roles to the level of Metallurgy Manager, being responsible for the processing plant and maintenance department at operations in Australia.

3. 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.

4. I am responsible for the preparation of the technical report entitled "B2Gold Corp, Fekola Gold Project, NI 43-101" dated July 24, 2015, with an effective date of June 30, 2015, (the "**Technical Report**"), with specific responsibility for Sections 1, 2, 3, 13, 17, 18 (except 18.4, 18.5 & 18.6), 19, 21, 24, 25, 26 and 27. I most recently visited the property on May 20, 2015 for 3 days.

5. I have not had prior involvement with the property that is the subject of this Technical Report.

6. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

7. I have read National Instrument 43-101 and Form 43-101, and the Sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible

contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 24th day of July, 2015

(Signed) "*Sandy Hunter*"

_____

Sandra (Sandy) Hunter MAusiMM(CP)

CERTIFICATE OF AUTHOR

I, Ken D. Jones, PE, do hereby certify that:

1. I am currently employed as Health, Safety, Environment and Permitting Manager by B2Gold Corp. ("**B2Gold**") with an office at Suite 3100, 595 Burrard Street, PO Box 49143, Vancouver, BC V7X 1J1.

2. I am a registered Professional Engineer (#42718, Colorado, USA). I graduated from the University of Iowa in 2001 with a B. Sc. in Chemical Engineering. I have practiced my profession for 13 years. I have developed, conducted and/or directed environmental and social studies including baseline investigations; materials geochemical characterization; hydrologic, air and noise modeling; closure planning and costing; and environmental and social impact assessment for hard rock mining projects in over a dozen countries in North and South America, Africa and Asia. I have developed, implemented and maintained programs for engineering and administrative compliance regarding international environmental, health and safety regulations and best practices at gold projects in Nicaragua, Namibia, and the Philippines.

3. 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.

4. I am responsible for the preparation of the technical report entitled "B2Gold Corp NI 43-101 Technical Report Feasibility Study on the Fekola Gold Project in Mali" dated July 24, 2015, with an effective date of June 30, 2015 (the "**Technical Report**"), with specific responsibility for Section 20. I most recently visited the property on April 28, 2015 for 10 days.

5. I have not had prior involvement with the property that is the subject of this Technical Report.

6. I am not independent of the issuer.

7. I have read National Instrument 43-101 and Form 43-101, and the Sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 24th day of July, 2015

(Signed) "*Ken D. Jones*"

_____

Ken D. Jones, PE

CERTIFICATE OF AUTHOR

I, William (Bill) Lytle, PE, do hereby certify that:

1. I am currently employed as Vice President by B2Gold Corp ("**B2Gold'**) with an office at Suite 3100, 595 Burrard Street, PO Box 4914, Vancouver, BC V7X 1J1.

2. I am a member of the American Society of Civil Engineers (membership # 9134979). I graduated from Colorado State University in 1995 with a B. Sc. in Chemical Engineering, and in 2010 with a M. Sc. in Civil Engineering. I have practiced my profession for 20 years. I have been directly involved in the design, construction, start up (Russia and Namibia) and operation of gold projects in Russia, Nicaragua, and Namibia. I have also participated in studies and the execution of infrastructure projects including detailed construction planning and execution of major mining and infrastructure projects in Russia, Nicaragua, and Namibia. I have been involved in the preparation and evaluation of financial models for development and operational gold projects.

3. 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.

4. I am responsible for the preparation of the technical report entitled "B2Gold Corp NI 43-101 Technical Report Feasibility Study on the Fekola Gold Project in Mali" dated July 24, 2015, with an effective date of June 30, 2015 (the "**Technical Report**"), with specific responsibility for Sections 4, 5, 19, and 22. I most recently visited the property on June 2, 2015 for 3 days.

5. My previous involvement with the property that is the subject of this Technical Report consists of acting as a Qualified Person for the technical report on the Preliminary Economic Assessment issued for the Fekola Gold Project, filed August 2014.

6. I am not independent of the issuer.

7. I have read National Instrument 43-101 and Form 43-101, and the Sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 24th day of July, 2015

(Signed) "*Bill Lytle*"

_____

William (Bill) Lytle, PE

CERTIFICATE OF AUTHOR

I, Peter D. Montano, PE, do hereby certify that:

1. I am currently employed as a Sr. Project Engineer by B2GoldCorp ("**B2Gold**") with an office at Suite 3100, 595 Burrard Street, PO Box 49143, Vancouver, BCm V7X 1J1.

2. I am a registered Professional Engineer (#42745, Colorado, USA). I graduated from the Colorado School of Mines in 2004 with a B.Sc. in Engineering and a B.Sc. in Economics. I have practiced my profession for 11 years. I have been directly involved in the design, construction, start up and operation of gold projects in Nicaragua and Namibia. I have also participated in and contributed to studies for mining projects in Venezuela, El Salvador, and Australia. I have been involved in the preparation and evaluation of financial models for development and operational gold projects.

3. 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.

4. I am responsible for the preparation of the technical report entitled "B2Gold Corp NI 43-101 Technical Report Feasibility Study on the Fekola Gold Project in Mali" dated July 24, 2015, with an effective date of June 30, 2015, (the "**Technical Report**"), with specific responsibility for Sections 15, 16, 18 and 21. I most recently visited the property on December 7, 2014 for 5 days.

5. I have not had prior involvement with the property that is the subject of this Technical Report.

6. I am not independent of the issuer.

7. I have read National Instrument 43-101 and Form 43-101, and the Sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 24th day of July, 2015

(Signed) "*Peter D. Montano*"

_____

Peter D. Montano, PE

CERTIFICATE OF AUTHOR

I, David J. T. Morgan, PE do hereby certify that:

1. I am the Managing Director of Knight Piesold Pty Limited with an office at Level 1, 184 Adelaide Terrace, East Perth, Western Australia, 6004.

2. I graduated from the University of Manchester in 1980 with a BSc in Civil Engineering, and the University of Southampton in 1981 with an MSc in Irrigation Engineering. I am a member in good standing of the Australasian Institute of Mining and Metallurgy (Australasia, 202216) and a Chartered Professional Engineer and member of the Institution of Engineers Australia (Australia, 974219). My relevant experience includes: Project Director – Akyem Gold project; Project Director – Geita Gold Mine; and Project Director – Ahafo Gold Project.

3. 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.

4. I am responsible for the preparation of the technical report entitled "B2Gold Corp NI 43-101 Technical Report Feasibility Study on the Fekola Gold Project in Mali" dated July 24, 2015, with an effective date of June 30, 2015, (the "**Technical Report**"), with specific responsibility for Sections 18.4 (TSF), 18.5 (Surface Water), and 18.6 (Water Supply). I most recently visited the property on 10 – 12 November, 2013 for 3 days.

5. My previous involvement with the property that is the subject of this Technical Report consists of involvement in preparing previous technical reports on the Fekola Gold Project.

6. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101

7. I have read National Instrument 43-101 and Form 43-101, and the Sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 24th day of July, 2015

(Signed) "*David J. T. Morgan*"

_____

David J. T. Morgan, PE

Date and Signature Page

The effective date of this Technical Report titled “B2Gold Corp NI 43-101 Technical Report Feasibility Study on the Fekola Gold Project in Mali” is June 30, 2015. The report date is July 24, 2015. This Technical Report was prepared and signed by the following authors:

Original document signed and sealed by:

(Signed) “*Tom Garagan*”
_____

Tom Garagan, P.Geo. July 24, 2015

(Signed) “*Sandy Hunter*”
_____

Sandra (Sandy) Hunter, MAusIMM(CP) July 24, 2015

(Signed) “*Ken D. Jones*”
_____

Ken D. Jones, PE July 24, 2015

(Signed) “*Bill Lytle*”
_____

William (Bill) Lytle, PE July 24, 2015

(Signed) “*Peter D. Montano*”
_____

Peter D. Montano, PE July 24, 2015

(Signed) “*David J. T. Morgan*”
_____

David J. T. Morgan, PE July 24, 2015