NEXTSOURCE materials

MOLO PHASE 2 PRELIMINARY ECONOMIC ASSESSEMENT

National Instrument 43-101 Technical Report

On the Molo Graphite Project located near the village of Fotadrevo, in the Province of Toliara, Madagascar

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This report contains certain non-GAAP measures. The non-GAAP measures do not have any standardized meaning within IFRS and therefore may not be comparable to similar measures presented by other companies. These measures provide information that is customary in the mining industry and that is useful in evaluating the Project. This data should not be considered as a substitute for measures of performance prepared in accordance with IFRS.

TABLE OF CONTENTS

1 SUMMARY ... 22
1.1 Introduction ... 22
1.2 Project Location ... 23
1.3 Project Description ... 23
1.4 Summary of Financial Results ... 24
1.5 Property Description and Ownership ... 25
1.5.1 Property Description ... 25
1.5.2 Ownership ... 25
1.5.3 The Company's Royalties ... 26
1.5.4 Permits ... 27
1.6 Geologic Setting and Mineralization ... 28
1.7 Mineral Resource Estimate ... 28
1.8 Exploration ... 29
1.9 Mineral Reserve Estimate ... 30
1.10 Metallurgical Test Work ... 30
1.11 Recovery Methods ... 31
1.12 Infrastructure ... 33
1.13 Geotechnical ... 34
1.14 Concrete ... 34
1.15 Storm Water ... 34
1.16 Product Pricing ... 34
1.17 Logistics ... 35
1.18 CAPEX and OPEX ... 36
1.19 Economic Analysis ... 37
1.20 Environmental and Permitting ... 38
1.20.1 Environmental and Social Impact Assessment ... 39
1.21 Conclusions ... 40
1.21.1 Geology ... 40
1.21.2 Mining ... 40
1.21.3 Tailings ... 40
1.21.4 Risks ... 40
1.21.5 Permitting ... 40
1.21.6 Metallurgical Test Work ... 40
1.22 Recommendations ... 41
1.22.1 Geology ... 41
1.22.2 Mining Recommendations and Concluding Report ... 41

1.22.3 Metallurgical Test Work 41
1.22.4 Recovery Methods 41
1.22.5 Infrastructure 42
1.22.6 Water....42
1.22.7 Environmental, Social....42
1.22.8 Permitting....43
2 INTRODUCTION 43
2.1 The Issuer.... 43
2.2 The Technical Team 43
2.3 Report History.... 44
2.4 Terms of Reference and Purpose 45
2.5 Sources of Information 45
2.6 Personal Inspections.... 47
3 RELIANCE ON OTHER EXPERTS 48
4 PROPERTY DESCRIPTION AND LOCATION 49
4.1 Madagascar Overview 49
4.2 Country Overview.... 49
4.2.1 Government Policy and Outlook Regarding the Mining Industry 51
4.3 Location 51
4.4 Property Area 52
4.5 Mineral Tenure and Title 53
4.6 Royalties 54
4.7 Permits.... 54
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY 55
5.1 Access 55
5.2 Physiography 57
5.3 Climate.... 58
5.4 Local Resources and Infrastructure 58
5.5 Security.... 59
6 HISTORY.... 59
6.1 Property-Scale Exploration History.... 59
7 GEOLOGICAL SETTING AND MINERALISATION.... 60
7.1 Regional Geology.... 60
7.1.1 Tectonic History of Southern Madagascar 63
7.2 Regional Geology as it Relates to the Molo Deposit.... 65
7.2.1 Lithological Descriptions of Individual Rock Formations....67
7.2.2 Structural Geology 68

7.3 Molo Property Geology 68
8 DEPOSIT TYPES 72
8.1 Graphite Deposit Types 72
8.2 Graphite Mineralisation on Molo 73
9 EXPLORATION 73
9.1 Geological Mapping 73
9.2 Trenching 74
9.2.1 Trench Sampling 75
9.3 Ground Geophysical Surveying 77
9.4 Prospecting and Sampling 79
10 DRILLING 79
10.1 Diamond Drill Contractor and Logistics 80
10.1.1 Core Handling Procedures 81
10.1.2 Core Logging 81
10.1.3 Core Recovery 81
10.1.4 Core Photography 81
10.1.5 Collar Survey 82
10.1.6 Down Hole Surveys 83
10.1.7 Geotechnical Logging 83
10.1.8 Diamond Drill Core Sampling 83
10.1.9 Diamond Drill Results 84
11 SAMPLE PREPARATION, ANALYSES AND SECURITY 86
11.1 Sample Preparation, Assay and Analytical Procedures 86
11.2 QA/QC 87
11.2.1 Blanks 87
11.2.2 Standards 88
11.3 Field Duplicates 90
11.4 Relative Density 90
12 DATA VERIFICATION 91
12.1 Collar and Down Hole Surveys 91
12.2 Drill Logs 91
13 MINERAL PROCESSING AND METALLURGICAL TESTING 93
13.1 Introduction 93
13.2 Flowsheet Development – Mintek 94
13.3 Flowsheet Development – SGS 94
13.4 Pilot Plant Campaign 96
13.5 Optimization and Metallurgical Variability Program 100

13.5.1 Sample Selection 101
13.5.2 Comminution Testing 102
13.5.3 Optimization Flotation Program 103
13.6 Variability Flotation – Phase I 104
13.6.1 Optical Mineralogy 109
13.7 Variability Flotation – Phase II 110
13.7.1 Drill Hole Composites 110
13.7.2 Area Composites 112
13.8 Concentrate Upgrading Tests 114
13.8.1 Optical Mineralogy of Upgraded Concentrate 119
13.9 Total Carbon and Graphitic Carbon Assay Methods – SGS 120
13.10 Additional Testing 120
13.10.1 Thickening 121
13.10.2 Filtration Tests 121
13.10.3 Concentrate Drying 122
13.10.4 Wet and Dry Screening Tests 122
13.10.5 Additional comments 122
14 MINERAL RESOURCE ESTIMATES 123
14.1 Geological Database 123
14.1.1 Topography 123
14.1.2 Drill Holes 124
14.1.3 Relative Density 126
14.2 Geological Model on Which the Grade Estimation is Based 126
14.2.1 Grade Domaining 126
14.2.2 Grade Domaining using Leapfrog 127
14.2.3 Domaining in Datamine 129
14.3 Compositing 131
14.3.1 Composited Statistics 131
14.4 Variography 134
14.5 Top Capping 135
14.6 Grade Estimation 136
14.6.1 Method 136
14.6.2 Kriged Neighbourhood Analysis 136
14.6.3 Model Construction and Parameters 138
14.6.4 Kriging Parameters 139
14.7 Model Validation 139
14.8 Grade Distribution Plots 142
14.9 Resource Classification 143

14.10 Resource Tabulation ... 145
15 MINERAL RESERVE ESTIMATES ... 146
15.1 Mineral Reserves ... 146
15.2 Whittle Optimisation Shell ... 147
15.2.1 Final Pit Design ... 147
15.2.2 Mining Modifying Factors ... 148
15.3 Factors Affecting the Mineral Reserve Estimate ... 149
16 MINING METHODS ... 149
16.1 Open Pit Optimisation ... 150
16.1.1 Pit Slopes ... 151
16.1.2 Mining Dilution ... 151
16.1.3 Mining Recovery ... 152
16.1.4 Plant Recovery and Costs ... 152
16.1.5 Open Pit Final Design ... 152
16.2 Mine Planning ... 153
16.2.1 Open Pit Optimisation and Conceptual Mine Plan Undertaken for the Anticipated Phase 2 Production Levels 153
16.2.2 The Pit Optimisation Approach ... 153
16.3 Pit Optimisation Results ... 155
16.3.1 Scenario 1 ... 155
16.3.2 Scenario 2 ... 157
16.4 Mining Operations ... 160
16.4.1 Mining Production and Support Equipment ... 160
16.4.2 Labour ... 161
16.4.3 Drill and Blast Design ... 162
16.4.4 Mine Infrastructure ... 163
16.4.5 Explosives Storage ... 163
17 RECOVERY METHODS ... 164
17.1 Process Description ... 164
17.2 Ore Receiving and Crushing ... 165
17.3 Primary Milling and Flash flotation ... 166
17.4 Rougher Flotation ... 166
17.5 Primary Concentrate Cleaning ... 167
17.6 Fine Flake Cleaning ... 167
17.7 Attrition Cleaning ... 168
17.8 Secondary Attrition Cleaning ... 169
17.9 Final Tailings Handling ... 169
17.9.1 Tailings Handling and Disposal ... 169

17.9.2 Tailings Filtration and Disposal 170
17.10 Final Concentrate Handling 170
17.10.1 Concentrate Handling 170
17.10.2 Concentrate Drying and Bagging 170
17.11 Water Services 171
17.12 Air Services 172
17.12.1 Collector Storage and Dosing 172
17.13 Froth Storage and Dosing 173
17.14 Flocculant Mixing and Dosing 173
17.15 Coagulant Mixing and Dosing 174
18 PROJECT INFRASTRUCTURE 174
18.1 Mining Infrastructure 174
18.1.1 Mining Office 174
18.1.2 Plant and LDV Workshop 174
18.1.3 EMV Wash Bay 175
18.1.4 New EMV Workshop 175
18.1.5 ANFO Storage 175
18.1.6 Explosives Magazine 176
18.1.7 Access Road to Explosives Magazine 176
18.1.8 Main Haul Roads 176
18.1.9 ROM Tip Ramp 176
18.2 Plant Infrastructure 176
18.2.1 Plant Office 177
18.2.2 Plant and LDV Workshop 177
18.2.3 Laboratory 177
18.3 Raw Water Supply 177
18.4 Process Plant, Storm Water and Raw Water Supply Dams 177
18.4.1 Ablution Facilities 178
18.4.2 Reagent Stores 178
18.5 Shared Infrastructure and Services 178
18.5.1 Access Roads 179
18.5.2 Terraces and Bulk Earthworks 179
18.5.3 Gate House and Turnstile Access Control 179
18.5.4 Re-fuelling Station 179
18.5.5 Waste Water Treatment Plant 179
18.5.6 Potable Water Treatment Plant 180
18.5.7 Storm Water Drainage 180
18.5.8 Storm Water Dams 180

18.5.9 Consumer / Incomer Sub-station 180
18.5.10 Plant Sub-stations 181
18.5.11 Transformers 181
18.5.12 Fencing 181
18.5.13 Fire Water System, Reticulation and Storage 181
18.5.14 Potable Water Reticulation 182
18.5.15 Potable Water Storage Tank 182
18.5.16 Waste Water Reticulation 182
18.5.17 Strategic Spares Storage 182
18.5.18 Tyre Storage Yard 182
18.5.19 General Storage Yard 182
18.5.20 Bulk Fuel Storage Facility 183
18.6 Power Supply - 150 tKpa E&CI 183
18.7 Alternative Power Supply (IPP) 184
18.7.1 Plant, Mine and Infrastructure Power Supply 184
18.7.2 Plant, Mine and Infrastructure Power Supply Philosophy 184
18.7.3 Thermal Design Philosophy 185
18.7.4 Solar Design Philosophy 186
18.7.5 Average Daily Generation 186
18.7.6 Real Estate 187
18.8 Tariff Model and Carbon Savings 187
18.8.1 Expected Tariff 187
18.8.2 Typical Cost Scenarios to be Expected 188
18.8.3 Applicable Assumptions and Exclusions 189
18.8.4 Pricing Assumptions 189
18.8.5 Summary of Presented Tariff 189
18.9 Electrical, Control and Instrumentation Design 190
18.9.1 General 190
18.10 Standards, Acts and Regulations 190
18.11 Cable Racking 191
18.12 Electrical 191
18.12.1 System Voltages 191
18.12.2 MV Switchgear 192
18.12.3 Transformers 193
18.12.4 Fire Detection and Suppression 193
18.12.5 System Earthing 193
18.13 Motor Control Centres 194
18.13.1 Motors - 380VAC 195

18.13.2 Cables (Voltage Might Differ Between Sites.) 195
18.13.3 Lighting and Small Power 196
18.13.4 Electrical Field Equipment 197
18.13.5 Power Factor Correction (PFC) 197
18.14 Instrumentation Design and Drafting Software 197
18.15 Voltage Levels 197
18.16 Field Instrumentation and Remote Input / Output (Rio) Equipment 197
18.17 Instrument Cabling 198
18.18 PLC Panels and Junction Boxes 198
18.19 Plant Control System 199
18.20 Scada and Computer Systems 199
18.21 Networks 199
18.22 UPS Power 199
18.23 Level of Automation 200
18.24 Control Modes 200
18.25 Tailings Disposal and Storage Facility 200
18.26 Geotechnical Report Assessment and Founding Strategy 200
19 MARKET STUDIES AND CONTRACTS 202
19.1 Graphite Overview 202
19.2 Flake Graphite Market 202
19.3 Flake Graphite Pricing 205
19.4 Contracts and Agreements 207
20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT 208
20.1 Description of Available Information 208
20.2 Applicable Laws and Standards 209
20.3 Environmental and Social Sensitivities 209
20.3.1 Geographic Placement of Infrastructure 210
20.3.2 Water Supply 210
20.3.3 Electricity Supply 210
20.3.4 Tailings Storage Facility (TSF) 210
20.3.5 Transport Route and Shipping Port 211
20.4 Environmental and Social Impact Assessment and Management Plans 212
20.4.1 Preparation Phase 212
20.5 Construction Phase 213
20.6 Operational Phase 213
20.7 Decommissioning and Closure Phase 213
20.8 Waste and Water Management 214

20.8.1 Mine Waste Assessment 214
20.8.2 Surface Water Environment 214
20.8.3 Ground Water Environment 214
20.8.4 Water Management 215
20.9 Socio-Economics 215
20.10 Mine Rehabilitation and Closure 216
20.11 Permitting and Stakeholders 217
21 CAPITAL AND OPERATING COSTS 218
21.1 Capex 218
21.1.1 Basis Of Estimate 218
21.1.2 Base Date....220
21.1.3 Base Currency....220
21.1.4 Scope 220
21.1.5 Escalation (Inflation)....220
21.1.6 Transport 220
21.1.7 Overall Philosophy....220
21.1.8 Assumptions 221
21.1.9 Spares, First Fill and Consumables 221
21.2 OPEX 221
21.2.1 Process Plant Operating Cost Summary 221
22 ECONOMIC ANALYSIS.... 232
22.1 General 232
22.2 Assumptions 232
22.2.1 Product Price 232
22.2.2 Mineral Royalties....232
22.2.3 Technical Assumptions 233
22.2.4 Economic Assumptions 233
22.3 Financial Model and Results 234
22.4 Sensitivity Analysis.... 236
22.4.1 Spider Sensitivity Analysis 237
23 ADJACENT PROPERTIES 239
24 OTHER RELEVANT DATA AND INFORMATION.... 239
24.1 Report on the Hazard (Hazard Study 2) Study carried out on the Proposed Molo Graphite Plant in Madagascar 239
25 INTERPRETATIONS AND CONCLUSIONS 240
25.1 Geology.... 240
25.2 Mining.... 240
25.3 Tailings.... 240

25.4 Risks 241
25.5 Permitting 241
25.6 Metallurgical Test Work 241
26 RECOMMENDATIONS 241
26.1 Geology 241
26.2 Mining 241
26.3 Metallurgical Test Work 241
26.4 Recovery Methods 242
26.5 Infrastructure 242
26.6 Water 242
26.7 Environmental, Social 242
26.8 Permitting 243
27 REFERENCES 243
27.1 Geology 243
27.2 Mining 244
27.3 Process 244
27.4 Infrastructure 246
27.4.1 Plant Geotechnical Information 246
27.4.2 Water 246
27.4.3 Tailings and Plant 247
27.5 Market Studies 247
27.6 Environmental, Social, Permitting 248
28 CERTIFICATES OF AUTHORS 249

LIST OF TABLES

Table 1: Summary of Financial Results 24
Table 2: Mineral Resource Statement for the Molo Graphite Deposit - September 2014 28
Table 3: Mineral Reserves 30
Table 4: Metallurgical Data - Flake Size Distribution and Product Grade 31
Table 5: Pricing Matrix - Flake Size Distribution Grouping and Product Grade 31
Table 6: Capital Costs 36
Table 7: Operating Costs per Tonne of Finished Graphite Concentrate 37
Table 8: Economic Analysis of the Project 37
Table 9: Source of Information 45
Table 10: Historical Activities on the Project 59
Table 11: Samples taken from the 2012-2014 trench exploration programmes 76
Table 12: Collar Co-Ordinates, Drilled Length and Orientation for the Molo Drill Holes 84
Table 13: Size Fractions Analysis Results of Test F24 96
Table 14: Summary of Comminution Test Results for Pilot Plant Composites 98
Table 15: Summary of Mass Balance for Pilot Plant Test PP-17B 99
Table 16: Average Combined Concentrate from Fifteen Pilot Plant Surveys 99
Table 17: Comparison of Laboratory and Pilot Plant Metallurgical Results 100
Table 18: Sample for Optimization and Variability Testing 101
Table 19: Comminution Composites – SGS and Mintek 102
Table 20: Summary of Comminution Tests – SGS 102
Table 21: Summary of Comminution Test - Mintek 2014 102
Table 22: Variability Composites – Phase I 104
Table 23: Variability Composites – Phase II 110
Table 24: Drill Holes included in Area Composites 112
Table 25: Comparison of Flake Size Distribution 112
Table 26: Mass Recovery and Total Carbon Grades of Size Fractions of Final Concentrate 119
Table 27: Summary of Lithological Codes Used 124
Table 28: Summary of Kzone Flagging in Datamine 131
Table 29: Statistical Summary of C% Per Kzone 132
Table 30: Summary of Top Capping Values 136
Table 31: Statistical Comparison between Composited Samples and Model Estimates 140
Table 32: Mineral Resource Statement for the Molo Graphite Deposit – September 2014 146
Table 33: Molo Reserves Statement 149
Table 34: Whittle Input Parameters 151
Table 35: Pit Optimisation Parameters (Source Sound Mining 2022) 153
Table 36: Scenario 1 Pit Optimisation Results (Source: Sound Mining, 2022) 156
Table 37: Scenario 2 Pit Optimisation Results (Source: Sound Mining, 2022) 158
Table 38: Mining Production and Support Equipment 160

Table 39: Mining Labour Compliment for Phase 2 161
Table 40: Generation Operating and Maintenance Scenarios 186
Table 41: Cost Saving Scenarios 188
Table 42: Summary of Tariff 190
Table 43: Expected Allowable Bearing Capacity for Encountered Gneiss Bedrock 201
Table 44: 12 Month Flake Graphite Pricing With Carbon Contents Between 96-97% C and on A Fob China Basis 206
Table 45: Current (January 2022) flake graphite pricing for Molo distribution 207
Table 46: Information Summary 208
Table 47: Capital Costs 218
Table 48: Phase 2 processing plant capital costs 219
Table 49: Other capital costs 219
Table 50: Operating Costs Per Tonne of Graphite Concentrate 221
Table 51: Process Plant Opex Estimate 222
Table 52: Labour Costs 224
Table 53: Breakdown of the Operating Cost for Power 226
Table 54: Operating Costs for Water 227
Table 55: Reagent Operating Costs 227
Table 56: Milling Costs 229
Table 57: Product Drying Costs 229
Table 58: Final Tailings Handling Costs 229
Table 59:Maintenance Costs 230
Table 60: Laboratory Cost Breakdown 231
Table 61: Miscellaneous Costs 232
Table 62: Scenario 2 Technical Assumptions for Financial Analysis 233
Table 63: Economic Assumptions for Financial Analysis 233
Table 64: Financial Analysis Summary 234
Table 65: Financial Analysis Results by year 235
Table 66: Economic Metrics with Varying Discounts 236
Table 67: IRR Sensitivity 237

LIST OF FIGURES

Figure 1: Map of Madagascar 50

Figure 2: Molo Project Location 52

Figure 3: Molo Project - Exploitation Permit PE #39807 53

Figure 4: Road Access to the Molo Area from the Town of Toliara....................56

Figure 5: Airstrip At Fotadrevo 57

Figure 6: View of the Molo Project Area Showing the Nature of the Vegetation 58

Figure 7: Geological Map of Madagascar Showing the Distinctive Crystalline Basement and Sedimentary Basins in the West (Source: Besarie (1964) 61

Figure 8: Country Geology: Geological Blocks (source: AGP Mining Consultants (2011))....................62

Figure 9: Tectono-Metamorphic Units of the Precambrian Terrain of Madagascar (source: AGP Mining Consultants, 2011)63

Figure 10: Structural and Lithological Sketch Map of South-East Madagascar Showing the Positions of yhe Major Shear Zones (source: Rakotondrazafy et al.,2008) 65

Figure 11: Position of the Molo Project Area within the Overall Strain Pattern Documented for Southern Madagascar.......66

Figure 12: Map Showing the Surface Geology of the Molo Deposit 69

Figure 13: Outcrop Exposure of Graphitic Schist on the Molo Project....................70

Figure 14: Geological Model of the Molo Deposit Showing the Nature of the Two Tightly Oppressed 'High" Grade (Hg East and Hg West) Mineralized Zones....................70

Figure 15: Garnet Gneiss in the Footwall of Molo-4 at a Depth of 132m 71

Figure 16: Trenching for the Bulk Sample on Molo, May 2013 74

Figure 17: Plan Showing the Positions and Grades of All the Trenches Excavated on the Molo Deposit, Overlain on the Topographic Elevation Map....................75

Figure 18: EM31 Geophysical Survey Map of the Molo Deposit Showing the Positions of Holes Molo-01 to Molo-07 Drilled During 2011 78

Figure 19: Borehole Collar Positions for All of the Known Boreholes Drilled on the Molo Deposit....................80

Figure 20: Core Photographs of Borehole Molo-05 Core (Boxes 17 and 18) as Supplied to CCIC by the Company................82

Figure 21: Concrete Marker Showing the Collar Position of Borehole Molo 12-02 83

Figure 22: Plot of Log %C Versus the Date of the Analysis 2014 for the Blanks....................87

Figure 23: Original (Orig) Versus Duplicate (Dupe) Plots....................90

Figure 24: East-West Cross-Section Showing the Collar Positions with Respect to Topography 91

Figure 25: Borehole Fotsy-06 - One of the Bore Holes Check Logged during May 2012 Site Visit 92

Figure 26: Flowsheet for Molo Mineralization – Mintek....................94

Figure 27: Proposed Molo Mineralization Flowsheet – SGS 96

Figure 28: Map Showing Position of Bulk Sampling Trench 97

Figure 29: Molo Pilot Plant Flowsheet....................98

Figure 30: Optimized Flowsheet....................105

Figure 31: Range of Mass Recovery into Size Fractions (V1 to V32 and Pilot Plant) 106

Figure 32: Range of Carbon Grades of Size Fractions (V1 To V32 and Pilot Plant)....................107

Figure 33: Combined Concentrate Grade of Tests on Variability Composites (V1 To V32)....................107

Figure 34: Location and Performance of Phase I Variability Drill Holes 108

Figure 35: Intercalated Graphite 109

Figure 36: Location and Performance of Phase I Variability Drill Holes 111
Figure 37: Combined Concentrate Grades of Twenty-One Area Composites 113
Figure 38: Location and Performance of Area Composites 114
Figure 39: Stirred Media Mill Size Fraction Analysis – Mass Recovery 116
Figure 40: Attrition Scrubber Size Fraction Analysis – Mass Recovery 116
Figure 41: Concentrate Grades of Size Fractions – Stirred Media Mill 117
Figure 42: Molo Flowsheet Including Stirred Media Mill Circuit 118
Figure 43: Optical Photomicrographs (Ppl) From The U-8 SFA 3rd ClN CONC +48 Mesh 120
Figure 44: Topographic Contours; Elevations In MAMSL 123
Figure 45: Drill Hole Posting Plan Coloured on C (%) Grades. 124
Figure 46: Cross Section Showing the Grade Distribution (C%) Nn the Drill Holes and Trenches 125
Figure 47: Statistical Comparisons Between Diamond Drill Hole (Dd) and Trench (Th) Samples. 125
Figure 48: Histogram of Relative Density Readings within the Graphitic Gneiss 126
Figure 49: Histogram of C Distribution 127
Figure 50: Isometric View Showing "Low" and "High" Grade Domains 128
Figure 51: West-East Section Showing the "Low" and "High" Grade Domains 129
Figure 52: "High" and "Low" Grade Boundary Analysis 130
Figure 53: Cross-Section Showing the Kzone Flagging in the Block Model 130
Figure 54: Histogram of Sample Length Prior to and After Compositing 131
Figure 55: Sample Distributions for the "Low Grade" Domain. 133
Figure 56: Sample Distributions for the "High Grade East" Domain 133
Figure 57: Sample Distributions for the "High Grade West" Domain. 134
Figure 58: Variogram model for "Low Grade" domain – down hole and isotropic 134
Figure 59: Variogram model for "High Grade East" domain – down hole and isotropic. 135
Figure 60: Variogram Model for "High Grade West" Domain – Down Hole and Isotropic. 135
Figure 61: Plan Showing the Location for the Krige Optimizations 137
Figure 62: Parent cell size optimizations 138
Figure 63: Search Distance Optimizations 138
Figure 64: Section Showing Block Model Coding 139
Figure 65:Trend Analysis plot for %C, "low" grade domain 141
Figure 66:Trend Analysis plot for %C, "High Grade East" domain 141
Figure 67: Trend Analysis plot for %C, "High Grade West" domain 142
Figure 68: Section Showing C (%) Grade Distribution in the Model 142
Figure 69: Grade Tonnage Curve for the "Low" grade domain 143
Figure 70: Grade Tonnage Curve for the "High" grade domains 143
Figure 71: Plan showing Mineral Resource Classification. 144
Figure 72: Section showing Mineral Resource Classification 145
Figure 73: Molo Final Pit Design 147
Figure 74: Molo Open Pit Mining Bench Layout 150

Figure 75: Isometric View of the Block Model (Source: Sound Mining, 2022) 154

Figure 76: Typical Section through the Block Model showing the Different Mineralised Zones (Source: Sound Mining, 2022) 155

Figure 77: Isometric View of the Selected Pit Shell for Scenario 1 (Source: Sound Mining, 2022) 156

Figure 78: Typical illustrative layout of EMV Workshop.....175

Figure 79: Typical Layout of Containerized Fuel Storage Tank.....183

Figure 80: Alternative Power Arrangement 185

Figure 81: Average Generation Profiles.....187

Figure 82: Hybrid Alternative Energy Plant – Real Estate.....187

Figure 83 Natural Graphite Production Volumes by Country (Data from Statista, 2020) 203

Figure 84: 2018 Estimated Production of 700,000 Tons of Natural Flake Graphite (Roskill, 2019) 204

Figure 85: Flake Graphite Feedstock Required to Supply the Lithium-Ion Anode Market is Projected to Grow From 300,000 Tpa in 2020 to 9,000,000 Tpa by 2040 (Benchmark, 2021). 205

Figure 86: Monthly Flake Graphite Pricing for Various Flake Sizes and Carbon Contents (Benchmark, 2021) 206

Figure 87: Infrastructure Overlaid on Environmental and Social Sensitivities 211

Figure 88: Transport Route Alternatives to Fort Dauphin 212

Figure 89: Molo 235 Mtpa Graphite Project Staff Organogram 223

List of Graphs

Graph 1: Graph Showing Carbon Concentration as Analysed in Std 1C 88

Graph 2: Graph Showing Carbon Concentration as Analysed in GGC-01 89

Graph 3: Graph Showing Carbon Concentration as Analysed In GGC-07 89

Graph 4: Pit Optimisation Results showing the Selected Pit Shell 34 (Source: Sound Mining, 2022) 155

Graph 5: Conceptual LoM Production Schedule for Scenario 1 (Source: Sound Mining, 2022) 157

Graph 6: Conceptual LoM Production Schedule for Scenario 2 (Source: Sound Mining, 2022) 158

Graph 7: Pit Optimisation Results showing the Selected Pit Shell 36 (Source: Sound Mining, 2022) 159

Graph 8: Sensitivity of Project IRR (Pre-tax and royalties) to changes in Selling Price, OPEX and CAPEX 237

Graph 9: Sensitivity of Project IRR (Post-tax and royalties) to changes in Selling Price, OPEX and CAPEX 238

Graph 10: Sensitivity of Project NPV (Pre-tax and royalties) to changes in Selling Price, OPEX and CAPEX 238

Abbreviations, Symbols, Acronyms and Units of Measure	
ADT	Articulated Dump Truck
Ammonium Nitrate / Fuel Oil	ANFO
Agence de Promotion du Secteur Minier	APSM
Ariary Madagascan Currency	MGA
BESS	Battery Energy Storage Systems
Betsimisaraka Suture	BS
Bureau de Recherches Géologiques et Minières (France)	BRGM
Bureau du Cadastre Minier de Madagascar	BCMM
Canadian Institute of Mining	CIM
Caracle Creek International Consulting	CCIC
Carbon	C
Centimetres	cm
Closed Side Setting (Crusher)	CSS
Cubic metres	m^3
Cubic metres per hour	m^3/h
Cubic metres per minute	m^3/min
Degrees	°
Degrees Celsius	°C
Diamond Drill Hole	DDH
Directly On Line	DOL
Dry metric tonnes	Dmt
Dry metric tonnes per hour	dmt/h
Earth Moving Vehicle	EMV
Environmental Commitment Permit	RIM
Electric Vehicles	EV
Feasibility Study	FS
Feet	‘ or ft
Front End Loader	FEL
Free Air Delivery	FAD

Abbreviations, Symbols, Acronyms and Units of Measure	
Global Positioning System	GPS
Greater than	>
Greater than or equal to	≥
Grams	G
Grams per tonne	g/t
Hectares	Ha
High Density Polyethylene	HDPE
Hours	H
IPP	Independent Power Producer
Inductively Coupled Plasma (Assay)	ICP
Intermediate Bulk Container	IBC
Internal Diameter	ID
Joint Venture Agreement	JVA
Kilobar	Kbar
Kilograms	Kg
Kilograms per tonne	kg/t
Kilometres	Km
Kilovolt-amps	kVA
Less than	<
Less than or equal to	≤
Life of Mine	LoM
Madagascar Minerals & Resources SARL	MMR
Madagascar Ministry of Environment's Office National pour l'Environnement (the National Office for the Environment)	ONE
Mesh size (of screen or sieve)	mesh
Metres	M
Metres above mean sea level	mamsl
Metric tonnes	T
Methyl Isobutyl Carbinol	MIBC

Abbreviations, Symbols, Acronyms and Units of Measure	
Micrometres	µm
Millilitres	mL
Millimetres	Mm
Million tonnes	Mt
Million tonnes per annum	Mtpa
Million Watts	MW
Mine Rehabilitation and Closure Programme	MRCP
Mineral Reserve Estimate	MRE
National Environmental Action Plan	NEAP
National Instrument 43-101	NI-43-101
Net Smelter Royalty	NSR
Notified Maximum Demand	NMD
Non-sulphide Gangue	NSG
Organisation of Petroleum Exporting Countries	OPEC
Original Equipment Manufacturer	OEM
Particle size distribution where 80% of particles in a stream are larger than the size indicated	P_{80}
Parts per million	PPM
POCC	Point of Common Coupling
Pollution Control Dam	PCD
PEG Mining Consultants Inc.	PEG
Percent	%
Percent graphitic carbon	% C(g)
Percent total carbon	% C(t)
Preliminary Economic Assessment	PEA
Projet de Gouvernances des Ressources Minérales	PGRM program
Projet de Reforme du Secteur Minier	PRSM program
Qualified Persons	QPs
Quality Assurance / Quality Control	QA/QC

Abbreviations, Symbols, Acronyms and Units of Measure	
Rock Quality Description	RQD
Run of Mine	RoM
Special Advisory Committee	SAC
Spheronized and Purified Graphite	SPG
Square metres	m^2
Standard Operating Procedures	SOP
Stirred Media Detritor	SMD
Storm Water Management Plan	SWMP
Taiga Consultants Ltd.	TAIGA
Tailings Storage Facility	TSF
Thousand Pascals	kPa
Tonnes per annum	tpa
Thousand tonnes per annum	ktpa
Thousand tonnes per month	ktpm
Thousand Watts	kW
Three dimensional	3D
Tonnes (1000kg)	T
Tonnes per cubic metre	t/m^3
Toronto Stock Exchange	TSX
Universal Transverse Mercator	UTM
Uranium Star Minerals SARL	USM
Variable Speed Drive	VSD
Whole Rock Analysis	WRA
X-Ray Fluorescence	XRF

1 SUMMARY

1.1 Introduction

The Company is a mineral exploration and development company based in Toronto, Canada. The Company is currently focused on the development of its 100% owned, flagship Molo Graphite Mine Project.

The Molo deposit is situated 160 km south-east of the city of Toliara, in the Tulear region of south-western Madagascar. The deposit occurs in a sparsely populated, dry savannah grassland region, which has easy access via a network of seasonal secondary roads radiating outward from the village of Fotadrevo. Fotadrevo in turn has an all-weather airstrip and access to a road system that leads to the regional capital (and port city) of Toliara and the Port of Ehoala at Fort Dauphin via the RN10, or RN13.

Geologically, Molo is situated in the Bekily block (Tolagnaro-Ampanihy high grade metamorphic province) of southern Madagascar. The Molo deposit is underlain predominantly by moderately to highly metamorphosed and sheared graphitic (biotite, chlorite and garnet-rich) quartzo-feldspathic schists and gneisses, which are variably mineralised. Near surface rocks are oxidised, and saprolitic to a depth, usually of less than 5m.

Molo was one of several surficial graphite trends discovered by the Company, (then Energizer Resources) in late 2011, and announced in early January 2012. The deposit was originally drill tested in 2012, with an initial seven holes being completed. Resource delineation, drilling and trenching on Molo took place between May and November of 2012 and allowed for a maiden Indicated and Inferred Resource to be stated in early December of the same year. This maiden Mineral Resource Estimate (“MRE”) formed the basis for a PEA, which was undertaken by DRA Projects in 2013 (the “Molo 2013 PEA").

The positive outcome of the Molo 2013 PEA led the Company to undertake another phase of exploratory drilling and sampling in 2014, which was done under the supervision of CCIC. This phase of exploration was aimed at improving the geological confidence of the deposit and it’s contained mineral resources and included an additional 32 diamond drill holes (totalling 2,063m) and 9 trenches (totalling 1,876m).

CCIC were subsequently engaged to update the geological model and resource estimate. The entire database on which this new model and resource estimate is based contains 80 drill holes (totalling 11,660m) and 35 trenches (totalling 8,492m). This new resource formed the basis of the Molo 2015 Feasibility Study, which was based on 860 ktpa of ore processing capacity (the “Molo 2015 FS”).

In 2017, the Company released the results of an updated Molo Feasibility Study, which was based on ore processing capacity of 240 ktpa (the “Molo 2017 FS”).

On September 27, 2019, the Company reported the results of an updated Feasibility Study (“FS”) consisting of two phases: Phase 1 consisted of a fully operational and sustainable graphite mine with a permanent processing plant capable of processing 240,000 tpa of ore producing approximately 17,000 tpa of graphite concentrate per year over a 30-year life of mine, and Phase 2 consisted of a modular expansion to a production capacity of 45,000 tpa of graphite concentrate in Year 3.

On March 29, 2021, the Company announced the initiation of the construction process for Phase 1 of the Molo Graphite Mine with a processing plant capable of processing 240,000 tpa of ore producing approximately 17,000 tpa of graphite concentrate.

Anticipating the future demand for industrial minerals such as those held by the Company (Graphite and Vanadium) is complex. The demand for these minerals is, to a large extent, driven by the development of the battery market which remains uncertain, but bullish. Significant research has been completed by various analysts and the consensus view is that an explosive increase in demand can be expected. The uncertainty, however, is the timing of such increase in demand.

The Company has announced graphite concentrate offtakes with a Japanese Trader and with *thyssenkrupp*. The Company is in the process of formalizing additional sales agreements. To ensure that the Company remains ahead of the competition and to appropriately plan for future market demand, the Company has opted for a flexible development approach, which comprises a modular solution yielding optimal cashflow and return metrics with suitable flexibility to enable them to rapidly respond to market changes.

As such, the Company requested the completion of a PEA-level study for an enhanced Phase 2 expansion.

This Technical Report (hereinafter referred to as the “PEA”) considers a Phase 2 stand-alone processing plant capable of processing 2,500,000 tpa of ore producing approximately 150,000 tpa of graphite concentrate over a 26 year LoM.

This PEA utilises the knowledge base of the FS technical report. Where applicable and relevant, Phase 2 amounts from the FS are updated for inflation and current market realities. Phase 2 costing is derived from current market pricing and the factorisation of pricing obtained during construction of Phase 1. Phase 2 costs are therefore deemed accurate to PEA level. Costs are expected to be further optimised through economies of scale which are not considered in this report.

The Company has every intention to develop Phase 2 in close succession to the completion of Phase 1 and has the mineral resources to support further increases of its mining and beneficiation capacity as the inevitable increase in demand is realised.

1.2 Project Location

The Molo deposit is located some 160 km south-east of Madagascar’s administrative capital (and port city) of Toliara, in the Tulear region and about 220 km NW of Fort Dauphin and is approximately 13 km NE of the local village of Fotadrevo.

1.3 Project Description

The Phase 1 of the Molo Graphite Mine consists of the construction of a greenfield open pit mine, a processing plant capable of processing 240,000 tpa of ore producing 17,000 tpa of graphite concentrate, and all supporting infrastructure including water, fuel, power, tailings (co-disposed), buildings and permanent accommodation. This PEA considers a stand-alone Phase 2 processing plant capable of processing 2,500,000 tpa of ore producing 150,000 tpa of graphite concentrate over a 26 year LoM, and all supporting infrastructure including water, fuel, power, tailings (co-disposed), buildings and permanent accommodation. Only the Phase 2 revenues, operating costs and capital cost estimates form the basis of the PEA financial model.

1.4 Summary of Financial Results

The financial results for Phase 2, consisting of a stand-alone processing plant capable of producing 150,000 tpa of graphite concentrate over a 26 year LoM, are summarized in Table 1 below. The results are based on a discounted cash flow analysis of Phase 2 using real cash flows, which do not include the effects of inflation. NextSource completed all financial modelling and sensitivity analysis, including the estimates for the NPV, IRR, payback and initial working capital.

Table 1: Summary of Financial Results

Description	***Phase 2 PEA (150K tpa production)***
Economic Highlights	
Pre-tax Net Present Value ("NPV") (8% discount rate) (1)(2)(4)(8)	*US$904.8 million*
Post-tax NPV (8% discount rate) (1)(2)(3)(4)(8)	*US$593.0 million*
Pre-tax Internal Rate of Return ("IRR") (1)(2)(4)(8)	*40.4%*
Post-tax IRR (1)(2)(3)(4)(8)	*31.4%*
Life of Mine ("LoM")	*26 years*
Pre-tax payback (1)(2)(4)(8)	*3.18 years*
Post-tax payback (1)(2)(3)(4)(8)	*3.74 years*
Capital costs ("CAPEX") including contingency of $31.96 million (2)	*US$155.8 million*
Initial working capital	*US$20.9 million*
Sustaining and closure CAPEX	*US$24.5 million*
Operational Highlights	
Graphite concentrate sale price (US$ per tonne of concentrate)(8)	*US$1,230.50*
Average operating costs FOB ("Opex") (US$ per tonne of concentrate following ramp-up)(7)	*US$495.62*
Average annual production of concentrate (5)(6)	*150,000 tpa*
Average ore mined per annum over LoM	*2,532,345 tpa*
Average head grade	*6.16%*
Concentrate purity (Cg) of finished product	*97%*
Average stripping ratio	*0.53:1*
Average carbon recovery	*88.30%*

Notes

(1) Assumes Project is financed with 100% equity.

(2) Capex includes process equipment, civil and infrastructure, mining, buildings, electrical infrastructure, project and construction services.

(3) Assumes 2% government gross revenue royalty, 3% Vision Blue gross revenue royalty, 1.5% NSR royalty and corporate tax rate of 20%.

(4) Assumes no inflationary adjustments in sales price, or operating costs.

(5) Assumes all mineralized material from the Company's 2019 Feasibility Study, including ore from the Measured, Indicated and Inferred Mineral Resource categories, are sent to the treatment plant.

(6) Assumes a cut-off grade of 4.5% carbon has been applied, with all material below this cut-off grade treated as waste.

(7) Assumes all concentrate will be sold on a FOB basis at the Port of Ehoala, Madagascar.

(8) Based on current market prices provided by UK-based commodity price reporting agencies Benchmark Minerals Intelligence and fast markets.

1.5 Property Description and Ownership

1.5.1 Property Description

The Project includes 790 claims and an area totalling 308.6 km^2.

The Project is centred on UTM coordinates 495,289 easting 7,345,473 northing (UTM 38S, WGS 84 datum), and is located 11.5 km east-north-east of the town of Fotadrevo.

The property is within Exploitation / Mining Permit PE #39807 which covers an area of 175 km^2 or 17,500 hectares ("ha"), and Exploration Permits PR #39806 and PR #39810 which cover areas of 96.1 km^2 (9609 ha) and 37.5 km^2 (3750 ha), respectively.

1.5.2 Ownership

On December 14, 2011, the Company entered into a Definitive JVA with Malagasy Minerals Limited, (hereinafter referred to as "Malagasy"), a public company on the Australian Stock Exchange, to acquire a 75% interest to explore and develop a group of industrial minerals, including graphite, vanadium and approximately 25 other minerals. On October 24, 2013, the Company signed a MOU with Malagasy to acquire the remaining 25% interest in the land position.

On April 16, 2014, the Company signed a Sale and Purchase Agreement and a Mineral Rights Agreement with Malagasy to acquire the remaining 25% interest. Malagasy retains a 1.5% Net Smelter Return Royalty ("NSR").

CCIC reviewed a copy of the Contrat d'Amodiation pertaining to this right and are satisfied that the rights to explore this permit have been ceded to the Company, or one of its Madagascar subsidiaries.

The Project was located within Exploration Permit PR #3432 as issued by the Bureau de Cadastre Minier de Madagascar ("BCMM") pursuant to the Mining Code 1999 (as amended) and its implementing decrees. On January 18, 2019, Permit PR #3432 was transformed into two Exploration Permits (PR #39806 and PR #39810) and an Exploitation Permit (PE #39807) by the Ministry of Mines, with the official permit being granted to the Company by the BCMM on February 14, 2019.

Mineral Resources and Reserves delineated in Sections 14 and 15 of this Report are entirely within the bounds of Exploitation Permit PE #39807. The Company holds the exclusive right to exploit / mine and explore for graphite within this license area for a period of 40 years and can renew the license several times for a further period of 20 years upon each renewal.

The Company holds the exclusive right to explore for a defined group of industrial minerals within Exploration Permits PR #39806 and PR #39810. These industrial minerals include the following: Vanadium, Lithium, Aggregates, Alunite, Barite, Bentonite, Vermiculite, Carbonatites, Corundum, Dimensional stone, (excluding labradorite), Feldspar (excluding labradorite), Fluorspar, Granite, Graphite, Gypsum, Kaolin, Kyanite, Limestone / Dolomite, Marble, Mica, Olivine, Perlite, Phosphate, Potash–Potassium minerals, Pumice Quartz, Staurolite, Zeolites.

Companies in Madagascar first apply for an exploration mining permit with the BCMM, a government agency falling under the authority of the Minister of Mines. Permits under usual circumstances are generally issued within a month. The number of squares varies widely by claim number.

The updated Decret requires the payment of annual administration fees of Permits Research of 15,000 Ariary (MGA) for exploitation permits in years one and two. Annual fees increase by multiplying by a factor equivalent to the number of years (plus 1) that the company has held the permit. Exploration permits have an updated duration of five years, with the possibility of two renewals of an additional three years each. Payments of the administration fees are due each year on 31 March, along with the submission of an activity report. Each year the Company is required to pay a similar, although increasing amount to maintain the claims in good standing.

Reporting requirements of exploration activities carried out by the title holder on an Exploration Permit are minimal. A title holder must maintain a diary of events and record the names and dates present of persons active on the Project. In addition, a site plan with a scale between 1/100 and 1/10,000 showing “a map of the work completed” must be presented. CCIC is of the opinion that the Company is compliant in terms of its commitments under these reporting requirements.

The Project has not been legally surveyed; however, since all claim boundaries conform to the pre-determined rectilinear LaBorde Projection grid, these can be readily located on the ground by use of Global Positioning System (“GPS”) instruments. Most current GPS units and software packages do not, however, offer LaBorde among their available options and therefore defined shifts must be employed to display LaBorde data in the WGS 84 system. For convenience, all the Company’s positional data is collected in WGS 84, and if necessary, converted back to LaBorde.

1.5.3 The Company’s Royalties

The Madagascar government retains a 2% gross revenue royalty, Vision Blue Resources Limited (“Vision Blue”) retains a 3% gross revenue royalty, and Malagasy retains a 1.5% net smelter return royalty on the Project.

1.5.4 Permits

Exploitation Permit PE #39807 (175 km^2) and Exploration Permits PR #39806 and PR #39810 are held under the name of a subsidiary of the Company called ERG (Madagascar) Ltd. S.A.R.L.U. and were granted to the Company by the BCMM on February 14, 2019.

The Madagascar Ministry of Environment's Office National pour l'Environnement (the National Office for the Environment) or "ONE", granted the Company's Environmental License for Phase 1 of the Molo Graphite Mine on April 8, 2019, after reviewing the following Exploitation Permit PE #39807:

Environmental and Social Impact Assessment ("ESIA") and Relocation Action Plan ("RAP") to International Finance Corporation ("IFC") Performance and World Bank Standards.

Completion of local and regional stakeholder and community engagement, with overwhelming support from both the local community and local government, as well as regional government.

Signed agreements with all potentially affected land occupants to accept compensation for any affected crops and grazing land and relocation if needed.

Approved capital investment certification from the BCMM.

Receipt of Cahier des Charges Minière (mining specification) from the BCMM as pre-requisite to submitting the ESIA and RAP to ONE for review.

Successful completion of the ONE's technical evaluation process which consisted of a site visit and four separate community consultations.

Joint agreement and signature of the Cahier des Charges Environnementales (environmental specification) with the ONE.

Specific Environmental Management Plans (S/EMPs) are approved for the following Project components:

- Relocation Action Plan and Livelihood Restoration Plan for Phase 1.
- Thermal and Solar Self-Generation of Electricity for Phase 1.
- The Development of Roads and Pipelines for Phase 1.
- The Waste Management Plan for Phase 1.

The approval of the following additional Specific Environmental Management Plans (S/EMPs) is pending:

- The development of the Base Camp.
- The development of the Processing Plant and associated buildings and structures.
- The development of the Opencast Pit.

The approval of the following additional permit is pending:

- The Tree Removal Permit.
- Industrial Operating License.
- Building / Construction Permit.
- Long-term Land Lease.

- Agreement with the Port of Ehoala.

1.6 Geologic Setting and Mineralization

The Molo deposit occurs within the regional Ampanihy Shear Zone. The most conspicuous feature of rocks found within this shear zone is their well-developed north-south foliation and vertical to sub-vertical nature. Martelat et al. (2000) state that this observed bulk strain pattern is clearly related to a transpressional regime during bulk horizontal shortening of heated crust, which resulted in the exhumation of lower crustal material.

The Project area is underlain by supracrustal and plutonic rocks of late Neoproterozoic age that were metamorphosed under upper amphibolite facies and deformed with upright north-northeast-trending structures. The supracrustal rocks involve migmatitic (± biotite, garnet) quartzo-feldspathic gneiss, marble, chert, quartzite, and amphibolite gneiss. The metaplutonic rocks include migmatitic (± hornblende / diopside, biotite, garnet) feldspathic gneiss of monzodioritic to syenitic composition, biotite granodiorite, and leucogranite.

1.7 Mineral Resource Estimate

Mineral Reserves and Mineral Resources did not change as a result of the PEA and remain the same as disclosed in the FS.

The Project hosts the following resources:

- Measured mineral resource of 23.62 Mt grading 6.32% Carbon ("C").
- Indicated mineral resource of 76.75 Mt grading 6.25% C.
- Inferred mineral resource of 40.91 Mt at 5.78% C.

The effective date of the Mineral Resource tabulation is August 14, 2014. The Mineral Resources are classified according to the Canadian Institute of Mining, Metallurgy and Petroleum definitions. A cut-off grade of 4% C was used for the “higher grade” zones and 2% C for the “lower grade” zones. It is important to note that while the ‘high’ grade resource occurs within the ‘low’ grade resource, each was estimated and reported separately.

A relative density of 2.36t per cubic meter was assigned to the mineralized zones for the resource estimation. The resource remains open along strike and to depth. The Mineral Resources above are inclusive of the Mineral Reserves below.

The current MRE for Molo is summarized in Table 2 below. The mineral resources are classified in the Measured Indicated and Inferred categories as defined by the Canadian Institute of Mining, Metallurgy and Petroleum definition standards.

Table 2: Mineral Resource Statement for the Molo Graphite Deposit - September 2014

Classification	Material Type	Tonnes	Grade - C%	Graphite - T
Measured	"Low Grade"	13 048 373	4.64	605 082
Measured	"High Grade"	10 573 137	8.4	887 835

Classification	Material Type	Tonnes	Grade - C%	Graphite - T
Total Measured		23 621 510	6.32	1 492 916
Indicated	"Low Grade"	39 539 403	4.73	1 871 075
Indicated	"High Grade"	37 206 550	7.86	2 925 266
Total Indicated		76 745 953	6.25	4 796 341
Measured + Indicated	"Low Grade"	52 587 776	4.71	2 476 157
Measured + Indicated	"High Grade"	47 779 687	7.98	3 813 101
Total Measured + Indicated		100 367 464	6.27	6 289 257
Inferred	"Low Grade"	24 233 267	4.46	1 080 677
Inferred	"High Grade"	16 681 453	7.70	1 285 039
Total Inferred		40 914 721	5.78	2 365 716

C% = carbon percentage; Graphite – T = Tonnes of graphite

Notes:

- Mineral Resources are classified according to the Canadian Institute of Mining definitions.
- Mineral Resources are reported Inclusive of Mineral Reserves.
- “Low Grade” Resources are stated at a cut-off grade of 2% C.
- “High grade” Resources are stated at a cut-off grade of 4% C.
- Eastern and western high-grade assays are capped at 15% C.

A relative density of 2.36t per cubic meter (t/m^3) was assigned to the mineralized zones for the resource tonnage estimation.

The total Measured and Indicated Resource is estimated at 100.37 Mt, grading at 6.27% carbon. Additionally, an Inferred Resource of 40.91 Mt, grading at 5.78% carbon is stated. When compared to November 2012 resource statement, (Hancox and Subramani, 2013), this shows a 13.7% increase in tonnage, a 3.4% decrease in grade and a 9.8% increase in graphite content.

The reason for the increase in tonnage is due to the 2014 drilling on the previously untested north-eastern limb of the deposit, which added additional new resources. Additionally, 23.62 million tonnes, grading at 6.32% carbon, have been upgraded by infill drilling from the Indicated to Measured Resource category.

1.8 Exploration

No further exploration is currently planned.

1.9 Mineral Reserve Estimate

Mineral Reserves and Mineral Resources did not change as a result of the PEA and remain the same as disclosed in the FS.

The mineral reserves referenced in this PEA are as per Table 3 below.

Table 3: Mineral Reserves

Category	Tonnage	C Grade (%)
Proven	14 169 741	7.00
Probable	8 266 944	7.04
Proven and Probable	22 436 685	7.02

Proven reserves are reported as the measured resources inside the designed open pit and above the grade cut-off of 4.5% C. Similarly, the probable reserves are reported as the Indicated Resources inside the designed open pit and above the grade cut-off of 4.5% C.

1.10 Metallurgical Test Work

The PEA is based on a full suite of metallurgical test work performed by SGS Canada Metallurgical Services Inc. in Lakefield, Ontario, Canada for the FS and remains the same for this PEA.

These tests included laboratory scale metallurgical work and a 200-tonne bulk sample / pilot plant program. The laboratory scale work included comminution tests, process development and optimization tests, variability flotation, and concentrate upgrading tests. Comminution test results place the Molo ore into the very soft to soft category with low abrasivity. A simple reagent regime consists of fuel oil number 2 and methyl isobutyl carbinol at dosages of approximately 120 g/t and 195 g/t, respectively. A total of approximately 150 open circuit and locked cycle flotation tests were completed on almost 70 composites as part of the process development, optimization, and variability flotation program.

The metallurgical programs culminated in a process flowsheet that is capable of treating the Molo ore using proven mineral processing techniques and its robustness has been successfully demonstrated in the laboratory and pilot plant campaigns.

The metallurgical programs indicated that variability exists with regards to the metallurgical response of the ore across the deposit, which resulted in a range of concentrate grades between 88.8% total carbon and 97.8% total carbon. Optical mineralogy on representative concentrate samples identified inter-layered graphite and non-sulphide gangue minerals as the primary source of impurities. The process risk that was created by the ore variability was mitigated with the design of an upgrading circuit, which improved the grade of a concentrate representing the average mill product of the first five years of operation from 92.1% total carbon to 97.1% total carbon.

The overall graphitic carbon recovery into the final concentrate is 88.3%.

The average composition of the combined concentrate grade is presented in Table 4. The size fraction analysis results were converted into a grouping reflecting a typical pricing matrix, which is shown in Table 5.

All assays were completed using control quality analysis and cross checks were completed during the mass balancing process to verify that the results were within the estimated measurement uncertainly of up to 1.7% relative for graphite concentrate grades greater than 90% total carbon.

Table 4: Metallurgical Data - Flake Size Distribution and Product Grade

Product Size	% Distribution	Product Grade (%) Carbon
+48 mesh (jumbo flake)	23.6	96.9
+65 mesh (coarse flake)	14.6	97.1
+80 mesh (large flake)	8.2	97.0
+100 mesh (medium flake)	6.9	97.3
+150 mesh (medium flake)	15.5	98.1
+200 mesh (small flake)	10.1	98.1
-200 mesh (fine flake)	21.1	97.5

Table 5: Pricing Matrix - Flake Size Distribution Grouping and Product Grade

Product Size	% Distribution	Product Grade (%) Carbon
>50 mesh	23.6	96.9
-50 to +80 mesh	22.7	97.1
-80 to +100 mesh	6.9	97.2
-100 mesh	46.8	97.6

Vendor testing including solid-liquid separation of tailings and concentrate, screening and dewatering of concentrate, and drying of concentrate was completed successfully.

1.11 Recovery Methods

The process design is based on an annual Phase 2 feed plant throughput capacity of 2,500,00 tpa at a nominal head grade of 6.16% C(t) producing an estimated average of 150,000 tpa of final concentrate.

The ore processing circuit consists of three stages of crushing which comprises jaw crushing in the primary circuit, followed by secondary cone crushing and tertiary cone crushing; the secondary and tertiary crushers operate in closed circuit with a double deck classification screen. Crushing is followed by primary milling and screening, graphite recovery by froth flotation and concentrate upgrading circuit by attritioning, and graphite product and tailings effluent handling unit operations. The crusher circuit is designed to operate 365 days per annum for 24 hours per day at ±55% utilization. The crushed product (P_{80} of approximately 13 mm) passes through a surge bin from where it is fed to the milling circuit.

The milling and flotation circuits are designed to operate 365 days per annum for 24 hours per day at 92% utilization. A single stage primary ball milling circuit is employed, incorporating a closed-circuit classifying screen and a scalping screen ahead of the mill. The scalping screen undersize feeds into a flash flotation cell before combining with the mill discharge material. Scalping and classification screen oversize are the fed to the primary mill.

Primary milling is followed by rougher flotation which, along with flash flotation, recovers graphite to concentrate from the mainstream. Rougher flotation employs six forced-draught trough cells. The recovered concentrate is then upgraded in the primary, fine-flake and attritioning cleaning circuits to an estimated final product grade of above 94% C(t). The primary cleaning circuit consists essentially of a dewatering screen, a polishing ball mill, a column flotation cell and flotation cleaner / cleaner scavenger trough cells.

The primary cleaner column cell concentrate gravitates to a 212 µm classifying screen, from where the large-flake oversize stream is pumped to a high-rate thickener located in the concentrate attritioning circuit whilst the undersize is pumped to the fine-flake cleaning circuit.

The fine flake cleaning circuit consists primarily of a dewatering screen, a polishing ball mill, a column flotation cell and flotation cleaner / cleaner scavenger trough cells. The attritioning cleaning circuit employs a high-rate thickener, an attritioning stirred media mill, a column flotation cell and flotation cleaner/cleaner scavenger trough cells. Fine flake column concentrate is combined with the +212 µm primary cleaner classifying screen oversize as it feeds the attritioning circuit thickener. Concentrate from the attrition circuit is pumped to the final concentrate thickener.

The combined fine flake cleaner concentrate and the +212 µm may also be processed through the secondary attrition circuit which consists of a dewatering screen, an attrition scrubber, column flotation cell and cleaner scavenger trough cells. Concentrate from this circuit is pumped to the final concentrate. The secondary attrition circuit is optimal.

Combined rougher and cleaner flotation final tailings are pumped to the final tailings thickener. Thickened final concentrate is pumped to a filter press for further dewatering before the filter cake is stockpiled prior to load and haul.

The concentrate thickener underflow is pumped to a linear belt filter for further dewatering and fed to a diesel fired rotary kiln for drying. The dried concentrate is then screened into four size fractions:

- +48 mesh.
- -48 + 80 mesh.
- -80 +100 mesh.
- -100 mesh.

The various product sizes are bagged and readied for shipping.

Chemical reagents are used throughout the froth flotation circuits and thickeners. Diesel fuel is used as collector and liquid Methyl Isobutyl Carbinol ("MIBC") is used as frother within the flotation circuits. Diesel collector is pumped from a diesel storage isotainer, from where it enters a manifold system which supplies multiple variable speed peristaltic pumps which discretely pump the collector at set rates to the various points-of-use within the flotation circuits.

MIBC frother is delivered by road to an isotainer. A manifold system on the storage isotainer supplies multiple variable speed peristaltic pumps, which discretely pump the frother at set rates to the various points-of-use within the flotation circuits.

Flocculant powder (Magnafloc 24) is delivered by road to the plant reagent store in 25 kg bags. The bags are collected by forklift as required and delivered to a flocculant mixing and dosing area. Here the flocculant is diluted as required using parallel, duplicate vendor package automated make-up plants, each one being dedicated to supplying the concentrate and tailings thickeners due to the flocculant types required being different for each application. Variable speed peristaltic pumps discretely pump the flocculant at set rates to the thickeners' points-of-use.

Coagulant powder (Magnafloc 1707) for thickening enhancement is handled similarly to the flocculant as described above, the exception being that a single make-up system is provided to supply both the concentrate and tailings thickeners. Again, variable speed peristaltic pumps discretely pump the coagulant at set rates to the thickeners' points-of-use.

1.12 Infrastructure

The Project is located in a relatively remote part of south-western Madagascar, approximately 13 km north-east of the local village of Fotadrevo. There is currently limited infrastructure on site and although some infrastructure will be built during Phase 1 of the Project additional infrastructure will be required. The following Project infrastructure will be required and/or will have to be upgraded.

The following elements are all part of the Project scope:

- Raw water supply will have to be increase from 3 to 21 bore holes extracting ground water.
- Power supply will have to be increased.
- Sanitation for the plant, permanent camp will have to be expanded.
- Storm water control and management will have to be expanded.
- All permanent buildings, (offices, workshops, ablution facilities), will have to be duplicated.
- A new mining workshop, product warehouse and reagent storage will have to be constructed.
- In plant roads.
- Haul road.
- Waste, high and low grade -Rock dumps.

Although the Phase1 plant is a small plant, some of the facilities in this plant could be leveraged, which will make construction of the Phase 2 plant significantly easier.

1.13 Geotechnical

The geotechnical investigation conducted by SRK Consulting in 2014 was used as reference document for the design and planning of this Phase of the Project. (Report 479297 / Plant Geotech / Final). This was augmented with reports from both Jones and Wagener (TN162-21-J335-00-Rev 0) and C.E.R.MAD (report number CER 21 10 Soil Stabilization) during construction of Phase 1.

In summary, transported soils are present across all areas investigated to shallow depths not exceeding a maximum depth of 0.6m. From the consistencies noted during test pit excavations the transported soils are anticipated to have a maximum allowable bearing capacity of 100 kPa, limiting total consolidation settlement to 25 mm.

Residual soils were noted in the majority of the test pits excavated and comprised dense to very dense silty and/or clayey sands. The residual soils are expected to have a maximum allowable bearing capacity of 200 kPa, limiting total consolidation settlement to 25 mm (differential settlement expected to be half this value). This could only be achieved though soil stabilisation with 5% cement added to the in-situ material.

As rock is located at a shallow depth at most locations it is recommended that structures generally be founded on rock rather than the overlying thin soils. However, light structures with loads of less than 100 kPa could be founded on the soils if necessary.

A suitable source of both backfills and aggregates need to be identified for Phase 2 of the Project.

1.14 Concrete

Concrete grades and mix design were selected taking into consideration durability requirements. Particular attention will be given to wet process plant areas and wash down slabs. All foundations were designed as pad, or raft type foundations with load bearing pressures not exceeding 150 kPa. Foundations were designed to minimize settlement.

1.15 Storm Water

Storm water that run-off within the process plant areas will be dealt with by a minimum slope on the terrace platform. Run-off is then collected in concrete lined V-drains.

Storm water within the process plant area will be collected though dedicated storm water containment channels and then handled accordingly.

1.16 Product Pricing

As an industrial mineral, flake graphite pricing is determined by three factors:

- Flake size.
- Carbon purity.
- Industry-specific technical attributes of the flakes (Benchmark, 2017a; Roskill, 2017).

Flake sizing is broadly classified into four ranges:

- Small (-100 mesh, or <75 µm)

- Medium (-80 to 100 mesh, or 75 µm to 180 µm)
- Large (-50 to 80 mesh, or 180 µm to 300 µm)
- Extra-large, or jumbo (+50 mesh, or >300 µm).

These flake sizes are in turn classified by carbon content (”C”) and are typically sold in ranges of 88% to 93% C, 94% to 95% C and 95% to 97% C. The specific technical attributes of the flakes are then defined by end-user parameters such as expansion co-efficient, thermal and electrical conductivity and charge-discharge stability and efficiency. As the technical parameters sought by end-users are proprietary to their processes, pricing is not publicly available.

The selling price of US$1,230.50 as outlined in Section 19.3 is the volume weighted average sales price for the various flake sizes and grades of graphite concentrate that are expected to be produced from the Molo deposit. This price used was based on current market prices provided by UK-based, commodity price reporting agencies Benchmark Minerals Intelligence and Fastmarkets, who are recognized as leaders in providing independent and unbiased market research, pricing trends and demand and supply analysis for the natural flake graphite market.

1.17 Logistics

The Port of Ehoala at Fort Dauphin is a modern (2009) port developed by Rio Tinto for the QMM Project. It has a 15m draft with shipping lines calling on a regular basis. There are, however, no crane facilities and vessels require their own cranes.

The following equipment is available at the port.

1 x 3.5T Telehandler.

5 x Trailers for container movement (2 x 40 ft, 3 x 20 ft).

1 x Tractor.

3 x Reach stackers 45T Capacity.

6 x Forklifts (1 x 2.5T; 2 x 5T; 3 x 7T).

The port is fenced and there is a security service (G4S) for port guarding day and night. Despite the presence of a national airport, the port of Ehoala is mainly connected to the hinterland destinations by road. All types of trucks can obtain access to the port and this berth for cargo off-loading. However, the majority are container trucks (20 ft and 40 ft).

CMA CGM is currently the only carrier who offers a service into Fort Dauphin from South Africa. CMA has a small feeder vessel who services the route from Durban to Fort Dauphin. This vessel is limited to a maximum of 45 TEU’s per sailing. Sailings are every two weeks and for this reason, shippers are limited to 10 TEU’s per booking and only on special request with sufficient notice will this be reconsidered by CMA. It must also be noted that bookings made in advance is subject to a cancellation fee if the booked space is not occupied.

Customs are available on site and clearance can be streamlined via pre-clearance in order to lessen standing time of the containers once arrived. It is to be noted that all cargo items imported into the Republic of Madagascar, needs to have a BSC online cargo tracking note. Failing to submit the BSC certificate, cargo cannot be cleared, and the shipment will be sent back to origin and be subject to a fine of US$2,500 per bill of lading, plus regulations charges. All containers, vehicles, bulk commodities, including airfreight requires a BSC certificate.

The route from Molo to Fort Dauphin runs either via the RN 10, or the RN 13. Both these routes are in relatively poor condition and trucks are expected to take between four and five days to make the round trip. A truck was monitored over the route by a Madagascan trucking contractor to gauge cycle times and they managed to complete the journey in two days each way. This was in the dry season, in the wet season there may be periods of time when the roads become impassable. No money has been budgeted for roads repairs, or upgrades.

The rainy season in Madagascar starts is November to April, during this time transportation of cargo can be delayed for several weeks. Due to the poor road conditions, majority of cargo would have to be transported to site during the dry season. Cargo transport limitations include:

- 12m (L) x 3.5m (W) x 2.8m (H) at a maximum of 35T per 3-axle trailer.
- 12m (L) x 2.5m (W) x 3.5m (H) at a maximum of 26T per 2-axle trailer.

Cargo exceeding 4m width pose problems to transport due to the Manambaro Bridge, as there is no possibility to divert. Some access areas would also need to be adjusted for items holding a width of 2.3m to 3.6m. (Ex. Raft of Bevilana). Any cargo exceeding the above-mentioned limitations would have to be considered on a case-by-case basis prior to importation.

For cargo's exceeding 30T, a deviation to the right of Manambaro bridge is possible, however, this is subject to prior negotiations with the owners of the rice fields through which the possible deviation will have to run through.

Specialised trailers and equipment for transporting out-of-gauge items are limited. The design of equipment / plant would have to consider above mentioned limitations to ensure equipment can be transported to site from port.

1.18 CAPEX and OPEX

The Phase 2 construction costs are estimated at US$155,851,935 including a 25% contingency. An additional US$20,876,922 is estimated for initial working capital. Over the LoM, an additional US$24,500,00M is estimated for sustaining capital and closure costs for equipment replacement and rehabilitation of the site at the end of the Project. The base date for the capital costs is January 2022 and no provision has been made for inflation. The accuracy of capital costs is considered to be within ±25%.

Table 6 summarizes the capital cost requirements.

Table 6: Capital Costs

Capital Cost Breakdown	Phase 2 Costs
Supply Items (Plant & Logistics)	US$43,200,996
Non supply Items (Engineering & Management, Civil & Infrastructure, P&G, Mechanical erection)	US$80,692,483
Sub Total	***US$123,893,479***
Fees and Contingencies	US$31,958,455

Capital Cost Breakdown	Phase 2 Costs
Construction CAPEX Total	**US$155,851,934**
Additional costs: Initial working capital Sustaining costs over LoM and closure costs at end of project	 US$20,876,922 US$24,500.000

The average operating costs per tonne of graphite concentrate delivered on a FOB basis to the Port of Fort Dauphin, Madagascar, during the LoM is presented in Table 7.

Table 7: Operating Costs per Tonne of Finished Graphite Concentrate

Breakdown	Phase 2 OPEX
Mining (US$/T)	US$145.88
Processing (US$/T)	US$206.75
Trucking to local port / Ft. Dauphin (US$/T)	US$133.00
General and Administration (US$/T)	US$10.00
Total	**US$495.62**

The reader should note that the estimated operating costs per tonne assume that the processing plant can successfully handle the variability in the ore body. As demonstrated by the SGS test work that is discussed in detail in Section 13, there is a risk that:

- The flake size distribution could be worse than expected.
- The product grade could be lower than expected.
- The recoveries could be lower than expected
- Combination of all of above.

If the plant does not perform as expected, this could have a material impact on operating costs.

1.19 Economic Analysis

The economic analysis of Phase 2 revenues, operating and capital costs over a 26 LoM using discounted cash flow methods is summarized in Table 8 below. All values are expressed in millions of US$.

Table 8: Economic Analysis of the Project

Metric	Values
Before tax and royalties	
Total Project Cash Flows [(1)(2)(4)(5)]	US$2,718.1 million

Metric	Values
NPV @ 8%	US$904.8 million
NPV @ 10%	US$716.7 million
NPV @ 12%	US$574.4 million
IRR	40.4%
Payback Period	3.18 years
After tax and royalties	
Total Project Cash Flows (1)(2)(3)(4)(5)	US$1,884.9 million
NPV @ 8%	US$593.0 million
NPV @ 10%	US$459.3 million
NPV @ 12%	US$358.2 million
IRR	31.4.0%
Payback Period	3.74 years

Note

(1) Assumes Project is financed with 100% equity.

(2) Capex includes process equipment, civil and infrastructure, mining, buildings, electrical infrastructure, project and construction services.

(3) Assumes 2% government gross revenue royalty, 3% Vision Blue gross revenue royalty, 1.5% NSR royalty and corporate tax rate of 20%.

(4) Assumes no inflationary adjustments in sales price, or operating costs.

(5) Based on current market prices provided by UK-based commodity price reporting agencies Benchmark Minerals Intelligence and fast markets.

1.20 Environmental and Permitting

The Madagascar Ministry of Environment's Office National pour l'Environnement (the National Office for the Environment) or "ONE", granted the Company its Environmental License for Phase 1 of the Molo Graphite Mine on April 8 2019 after reviewing the following:

- Exploitation Permit PE #39807.
- Environmental and Social Impact Assessment ("ESIA") and Relocation Action Plan ("RAP") to International Finance Corporation ("IFC") Performance and World Bank Standards.
- Completion of local and regional stakeholder and community engagement, with overwhelming support from both the local community and local government, as well as regional government.

- Signed agreements with all potentially affected land occupants to accept compensation for any affected crops and grazing land and relocation if needed.
- Approved capital investment certification from the BCMM.
- Receipt of Cahier des Charges Miniér (mining specification) from the BCMM as pre-requisite to submitting the ESIA & RAP to ONE for review.
- Successful completion of the ONE's technical evaluation process which consisted of a site visit and four separate community consultations.
- Joint agreement and signature of the Cahier des Charges Environnementales (environmental specification) with the ONE.

Specific Environmental Management Plans (S/EMP's) are approved for the following Project components:

- RAP and Livelihood Restoration Plan ("LSP") for Phase 1.
- Thermal and solar self-generation of electricity for Phase 1.
- The development of roads and pipelines for Phase 1.
- The waste management plan for Phase 1.

The approval of the following additional S/EMPs is pending:

- The development of the base camp.
- The development of the processing plant and associated buildings and structures.
- The development of the opencast pit.
- The approval of the following additional permits is pending.
- The Tree Removal Permit.
- Industrial Operating License.
- Building / Construction Permit.
- Long-term Land Lease.
- Agreement with the Port of Ehoala.

1.20.1 Environmental and Social Impact Assessment

A comprehensive Environmental and Social Impact Assessment was completed and submitted to Malagasy government as part of the Environmental Permit process.

Early integration of environmental and social sensitivities and risks ensured that the final impact assessment component revealed that there are no fatal flaws from an environmental and social perspective. The significance levels of impacts range from minor to major before any mitigation measures are applied and from minor to average with mitigation measures included. Notably, all major risks require significant reduction in risk via stringent controls. These controls have been incorporated into the Project design and planning with additional operational controls specified within the various environmental and social management plans.

To this end, the ESIA contains a chapter which details specific management measures which either remove the risks completely, or reduce their significance to an acceptable level.

In addition, each specific environmental and social component has a prescribed monitoring plan which will be followed during each Project developmental phase. This is aimed at monitoring compliance against various specifications such as the baseline environment and predicted impact removal and reduction measures.

1.21 Conclusions

1.21.1 Geology

The Company's 2011 exploration programme delineated a number of new graphitic trends in southern Madagascar. The resource delineation drilling undertaken during 2012-2014 focussed on only one of these, the Molo Deposit, and this has allowed for an Independent, CIM compliant, updated resource statement for the Molo deposit.

The total Measured and Indicated Resource is estimated at 100.37 Mt, grading at 6.27% C. Additionally, an Inferred Resource of 40.91 Mt, grading at 5.78% C is stated. When compared to the November 2012 resource statement (Hancox and Subramani, 2013), this shows a 13.7% increase in tonnage, a 3.4 % decrease in grade, and a 9.8% increase in graphite content. The reason for the increase in tonnage is due to the 2014 drilling on the previously untested north-eastern limb of the deposit, which added additional new resources. Additionally, 23.62 Mt, grading at 6.32% Carbon, have been upgraded by infill drilling from the Indicated to Measured Resource category.

1.21.2 Mining

The mineral reserves of 22,437,000t at an average grade of 7.02% were declared for the FS and is applied to this PEA.

1.21.3 Tailings

Tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining and on the same basis as Molo phase 1. In the next phase of the study a detailed design will be completed, complete with environmental and social impact assessment and closure.

1.21.4 Risks

In addition to the qualitative risk assessment completed during the Molo 2015 FS, a comprehensive HAZOP study was completed as part of the FS. The outcomes of that also informs this PEA study

1.21.5 Permitting

The Mining and Environmental Permits have been obtained for the Phase 1 Project, with most supplementary sectoral permits obtained, and the remainder pending final approval for Phase 1.

1.21.6 Metallurgical Test Work

Comprehensive metallurgical test programs culminated in a process flowsheet that can treat the Molo ore using conventional and established mineral processing techniques.

Process risks associated with the variability with regards to metallurgical performance have been mostly mitigated through the addition of an upgrading circuit. The upgrading circuit treated the combined concentrate after the secondary cleaning circuit. Reduced flake degradation and an improved process flexibility may be obtained by employing separate upgrading circuits for the coarse and fine flakes.

1.22 Recommendations

1.22.1 Geology

No further recommendations.

1.22.2 Mining Recommendations and Concluding Report

The mine planning scenarios described in this section have included Inferred Resources in the conceptual mine planning. While the Inferred Resource is included in the pit optimisation models, the percentage of the ore considered to be associated with an Inferred Resource is 14.6% in Scenario 1 and 7.2% in Scenario 2. This renders the Inferred Resource category as a minor contributor to the total mineable ore. To provide more confidence to the Mineral Resource estimate, infill drilling is recommended.

The open pit mining operations were planned by utilising both the unconstrained (Scenario 1) and constrained (Scenario 2) LoM production scenarios. Both scenarios indicate that a 150 ktpa carbon concentrate plant can be sustainably supported by the orebody over the planned LoM period.

1.22.3 Metallurgical Test Work

No further test work was carried out for this Phase of the study

Investigate the metallurgical impact of different attrition mill technologies such as stirred media mills, or attrition scrubbers.

Evaluate a range of different grinding media (e.g. different size, shape, material) to determine if flake degradation can be reduced without affecting the concentrate grade.

Develop a grinding energy versus concentrate grade relationship for the best grinding media. This will allow a more accurate prediction of the required attrition mill grinding energy as a function of the final concentrate grade.

Conduct attrition mill vendor tests to aid in the sizing of the equipment.

Carry out vendor testing on graphite tailings using the optimized reagent regime proposed by the reagent supplier.

Complete a series of flotation tests on samples covering mine life intervals for the PEA pit design.

1.22.4 Recovery Methods

No further recommendations.

1.22.5 Infrastructure

The following are recommended prior to the detailed design stage:

A detailed geotechnical investigation will need to be undertaken to identify and confirm suitable sources of concrete aggregate and backfill material. This testing will need to include for concrete material testing and the production of concrete trial mixes with the material identified

A detailed topographical survey will need to be undertaken of the proposed construction site, borrow pit areas and the access road between Fotadrevo and the mine site. This information is required prior to the final detailed design of the plant layout and associated earthworks

1.22.6 Water

The following is recommended during the detailed design phase:

Water quality and quantity data is required to provide a baseline for comparison once the Molo mine is commissioned. To provide the necessary baseline data, regular ground and surface water quality monitoring must be carried out leading up to the date when the Molo mine will be commissioned. Additional production and monitoring boreholes must be installed. This also should include the installation of flow meters on relevant pipelines to verify the dynamic water balance with measured flow rates during operations.

The installation of a weather station on the Project site should be done as soon as possible.

Quantitative and predictive water balance, ground water and geochemical analyses should be undertaken on regular intervals to update the water management plan.

1.22.7 Environmental, Social

The installation of a suitable weather station at/or as near as possible to the proposed Project site, even before construction commences, is recommended. Accurate, local weather data is almost non-existent in Madagascar. This data will prove invaluable for model calibration, improvement in baseline understanding and for future energy supply options which could utilize wind and/or solar power generation.

Clean and/or renewable energy supply should be considered as a medium to long term target.

Appointment of a community representative and the establishment of a mandate to sensitize the local communities prior to any Project activities.

Monitoring and auditing to commence at Project preparation phase.

Compilation of Standard Operating Procedures (“SOP”) for Environmental and Social aspects requiring direct management and intervention.

It is recommended that actual activity data, (e.g., kilometer’s travelled, or liters of diesel consumed) for a financial year is used when a GHG Assessment is being calculated.

Community recruitment, skills development and training should begin at Project preparation phase.

1.22.8 Permitting

Security of land tenure is a process and is estimated to take 6 to 9 months, thus this process should be commissioned as early as possible. The total area concerned is anticipated to be sufficient for the proposed Phase 2 expansion.

Application for all other necessary permits, (water use, construction, mineral processing, transportation, export, labour and so forth should be undertaken prior to the Phase 2 expansion.

Compilation of a comprehensive legal register.

Application for an amendment of the requisite environmental and construction approvals would be required for the Phase 2 expansion.

2 INTRODUCTION

2.1 The Issuer

This PEA report was prepared and issued by Erudite Strategies on behalf of NextSource Materials Inc.

2.2 The Technical Team

This Technical Report has been prepared by a combined technical team made up of people from the following organizations:

- Caracle Creek International Consulting (Proprietary) Limited – Geology.
- GCS Water and Environment (Pty) Ltd / Agetipa – Water.
- Globesight (Pty) Ltd - Environmental, Social, Permitting.
- SGS Lakefield – Metallurgical Test work.
- Metpro – Metallurgical Test work analysis.
- EPOCH – Tailings Storage Facility.
- Sound Mining (Pty) Ltd – Mining Design and Mining Capital and Operating costs.
- Erudite Projects – Earthworks, Civils, and Infrastructure.
- Erudite Projects (Pty) Ltd - Process Engineering. The process engineering was completed for the Molo 2017 FS by Met63 with the competent person being Mr. Paul Harvey supported by Mr A Mokwena. Neither of these experts are still employed by Met63 and as such were not able to provide further comment on the process engineering as part of the PEA. Mr Hector Mapheto, Pr.Eng who holds specific graphite processing knowledge and is currently employed by Erudite Projects (Pty) Ltd, has reviewed the process design and his comments are incorporated as appropriate.
- ISS, ASTRA, Velogic – Logistics
- OHMS – Mine Geotech
- RLH – Port Trade-off Study (From Molo 2015 FS)

2.3 Report History

The report history for the Project is as follows:

Date	Description of Event
February 2022	Finalisation of PEA report based on NI 43-101 headings
March 2021	Initiation of construction of Phase 1 of the Molo Graphite Mine
February 2021	Binding agreement with Vision Blue to provide financing for construction of Phase 1 of the Molo Graphite Mine
May 2019	Technical Report NI 43-101 FS report consisting of Phase 1 with processing capacity of 240 ktpa and Phase 2 expansion of 720 ktpa
April 2019	Environmental License for the Molo Graphite Project granted
February 2019	Official permit is granted to the Company by the BCMM
	Environmental and Social Impact Assessment completed and submitted to the Malagasy government
January 2019	Permit PR #3432 was transformed into two Exploration Permits (PR #39806 and PR #39810) and an Exploitation Permit (PE #39807) by the Ministry of Mines
July 2017	Molo 2017 FS (Detailed Engineering and Design) undertaken by Met 63
May 2017	Molo 2015 FS Technical Report NI 43-101 Technical Report issued by DRA
April 2017	Energizer Resources rebrands to NextSource Materials Incorporated
February 2015	Molo 2015 DFS undertaken by DRA
September 2014	Technical Report NI 43-101 issued by Hancox and Subramani
August 2014	Updated resource statement published
2014	Further exploratory drilling and sampling undertaken by CCIC – additional 32 diamond drill holes and 9 trenches
April 2014	Energizer signs a Sale and Purchase Agreement and a Mineral Rights Agreement with Malagasy to acquire the remaining 25% interest
April 2013	PEA Technical Report NI 43-101 issued
2013	PEA undertaken by DRA
October 2013	Energizer signs an MOU" with Malagasy to acquire the remaining 25% interest in the land position

Date	Description of Event
December 2012	Maiden resource statement published
May – November 2012	Resource delineation, drilling and trenching on Molo takes place
Early 2012	Seven holes drill-tested
January 2012	Energizer announces Molo discovery
December 2011	Energizer enters into a JVA with Malagasy Minerals Limited to acquire a 75% interest to explore and develop a group of industrial minerals
Late 2011	Molo discovered by Energizer
December 2009	Uranium Star Corporation rebrands to Energizer Resources
2007	Exploration works of the Molo graphite deposit completed by Uranium Star

2.4 Terms of Reference and Purpose

The terms of reference for this phase were to where applicable re-use the results of the FS, augment it with current actual data stemming from construction of Phase 1 of the Molo graphite mine and consider a stand-alone Phase 2 plant capacity to 2,500,00 tpa. This study is executed on a Preliminary Economic Assessment ("PEA") level to deliver Capital Cost Estimate ("CCE") (AACE Class 3 ±30%) for the proposed Molo Modular Graphite Plant Phase 2: 150 ktpa concentrate Project. The work covered the following key activities:

- To compile Capex and Operating Cost estimates for the Project
- To build a financial model and perform an economic analysis of the Project

2.5 Sources of Information

The following sources of information have been used in this Report. Refer Table 9 below.

Table 9: Source of Information

Report of Document	Author / Organization
Physical Data	
Site Topographical Data	The Company
Exploration Permit 3432	Malagasy Government
Civil Geotechnical Reports	EPOCH Consulting
Climate report	GCS Water and Environment
Geochemical analysis	GCS Water and Environment

Report of Document	Author / Organization
Geological Block Model	CCIC
Mine Geotech Report	OHMS
Mining and Reserves	
Mine Schedule	Erudite Projects
Equipment Tenders	Various
Process Plant and Metallurgy	
Process Design Criteria (PDC)	Erudite Strategies
Process Flow Diagrams (PFD)	Erudite Strategies
Mechanical Equipment List (MEL)	Erudite Strategies
Metallurgical Test Work Reports	SGS
Metallurgical Test Work Analysis	Metpro
Vendor Test Work Reports	Modular Supplier
Hazop Studies (Hazop 1 and Hazop 2 Reports)	The Company
Plant Layout Drawings	SYNCS
Block Plan	Erudite Strategies
Rheology Test Work	SGS
Production Schedule	Sound Mining
Tailings Disposal and Storage	
Site Selection Report	Erudite Projects (Pty) Ltd
TSF Report (including TSF Geotechnical Study)	Erudite Projects (Pty) Ltd
Infrastructure	
Site Plan	Erudite Strategies
Area Plan	Erudite Strategies
Infrastructure Equipment List (IEL)	Erudite Strategies
Plant and Camp Geotechnical Report	Erudite Projects (Pty) Ltd
Surface Water Supply Report - ANT_2 Dam	GCS
Power	
Power Plant Layout Drawings	Cross Boundary
Running Load Estimate	Erudite Strategies
Single Line Diagram	Erudite Strategies

Report of Document	Author / Organization
Vendor Quotes	Various
Fuel quotes	Various
Water	
Detailed Water Report	GCS Water and Environment
Water Balance	GCS Water and Environment
Environmental and Social	
Specialist Studies	GCS Water and Environment and Agetipa
ESIA (Environmental & Social Impact Assessment)	GCS Water and Environment and Agetipa
MOU (Memorandum of Understanding)	Agetipa
TOR (Terms of Reference)	Agetipa
Certification of Investment Amount	GCS Water and Environment and Agetipa
Sensitivity Maps	GCS Water and Environment
Permitting and Stakeholders	
Permit Register	Globesight
Stakeholder Register	Globesight
HR and Operational Readiness	
Malagasy Legislation	The Company
Malagasy Labour Rates	The Company
Staffing Plan	Erudite Strategies
Transport and Logistics	
Route Surveys	Astra, Velogic

2.6 Personal Inspections

The following personal inspections were done on the property for the original Molo Phase 1 execution Project:

Dr. Philip John Hancox between the 8 and 11 May, 2012. During this visit the graphite occurrences at Molo, Seta and Fotsy were investigated and various trenches on Molo were inspected. A second site visit was undertaken between the 19 and 21 May 2013, during the bulk sampling exercise. During this visit a number of collar beacons from the 2012 drilling campaign were inspected, as well as the trenching, logging and sampling methodologies.

Desmond Subramani visited the Molo site between the 15 and 19 February 2014, during the 2014 drilling campaign. The main aim of this visit was to plan the layout of the additional drilling and trenching required for the resource upgrade. This visit also covered the inspection of various borehole collars and open trenches, as well as a review of the drilling, logging and sampling procedures.

Dave Thompson visited the Molo site on 11 to 13 March 2014. The aim of this visit was to assess the site from a mining perspective.

Oliver Peters and John Stanbury have not visited the site.

P. Mogoera has not visited the site.

Although not an author of this report, a principal engineering geologist, Mr. Colin Wessels, visited the site from 5 to 13 April and 17 to 21 June 2014 to conduct a reconnaissance of the potential tailings dam sites and to supervise geotechnical investigations for the tailings dam, borrow pits and plant site.

The following staff from GCS although not QP's on the Project also spent time on site and contributed to the study in their various areas of expertise.

- Ferdi Pieterse : 11- 13 March 2014.
- Alkie Marais : 11 - 13 March 2014.
- Alistair Main : 11 - 14 March 2014.
- Alvar Koning : 11 - 15 March 2014 and 1 July – 12 July 2014.
- Hassen Khan : 11 - 15 March 2014 and 1 July – 12 July 2014.

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared for the Company and the information, conclusions, opinions and estimates contained herein are based on:

- Information available at the time of preparation of this report, (effective date being the 31 January, 2022).
- Assumptions, conditions and qualifications as set forth in this report.
- Data, reports and other information as supplied by the Company and other third-party sources.
- Experts involved in the run-up to the Molo Phase I execution and the execution currently in progress.

For the purpose of this report, the authors have relied on ownership information provided by the Company. In consideration of all legal aspects relating to the Project, Erudite Strategies and CCIC places reliance on the Company that the information relating to the legal aspects, and the status of surface and mineral rights, are accurate.

Property information in this report is sourced from previous works supplied by the Company and the authors are not responsible for the accuracy of any property data and do not make any claim, or state any opinion, as to the validity of the property disposition described herein.

For the preparation of this report, the authors relied on maps, documents and electronic files generated by the current and past exploration crews, contributing consultants and the technical team of the Company.

The Company has received specialist input from the following organizations in the preparation of this report and place full reliance on this information:

- EPOCH (Tailings Storage Facility).
- GCS Water and Environmental (Pty) Ltd and Agetipa (ESIA, Water, Geohydrology, Geochemistry, Water Balance)
- Globesight (Environmental, Social, Permitting).
- ASTRA, Velogic (Logistics).
- OHMS (Mine Geotechnical).
- Africa Business Solutions (Taxation).
- SGS Lakefield (Process analytics).
- Caracle Creek International (Resource definition).
- Erudite Projects (Pty) Ltd (Mining and Process).

This reliance applies to the following sections of the technical report as per competent persons sign off.

Any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Madagascar Overview

The Republic of Madagascar ("Madagascar") is not a traditional, or particularly well-known exploration and mining destination and is mostly under-explored, with the country subject to only very little modern era systematic exploration.

4.2 Country Overview

Madagascar is the largest island in the Indian Ocean (Figure 1) and the 4th largest island in the world (after Greenland, New Guinea and Borneo). It is located to the south-east of the African continent, from which it is separated by the Mozambique Channel. The country extends over 1,570 km from north to south, and is over 575 km wide, with a surface area of 587,040 km^2 and a 5,000 km long coastline.

Figure 1: Map of Madagascar

Source: http://goafrica.about.com/library/bl.mapfacts.madagascar.htm.

The capital of Madagascar is Antananarivo, a city of approximately 1,500,000 people, that is located in the central eastern area of the island approximately 150 km inland from the central-east coast, at an elevation of just over 1,200m above mean sea level ("mamsl").

Madagascar is officially bilingual, with French being the language of government and business. Malagasy (Malgache), a language of Malayo-Polynesian origin, is the official national language. English is taught in schools but is not widely spoken outside of business and government circles.

Madagascar is one of the world's hotspots of endemism and is recognised as one of the planets 17 recognized mega-biodiversity countries (Razafindralambo and Gaylord, 2005). Over 80% of the country's plant and animal species are unique to the island. Vegetation is varied and ranges from dense tropical rain forest in the east, Savannah in the central plateau and western coastal plain and spiny dry vegetation in the southern areas, which is where the Project is located.

4.2.1 Government Policy and Outlook Regarding the Mining Industry

The government of Madagascar embarked on an economic revival plan in 2000. At that time the Ministry of Energy and Mines ("Ministry") had already initiated reform through the Projet de Réforme du Secteur Minier ("PRSM") program, with the introduction of the new Mining Code in 1999, and the establishment of the Mining Titles (Cadastral) Registry (Bureau du Cadastre Minier de Madagascar, or BCMM) in 2000. These initiatives attracted new investors to Madagascar, including both junior and senior mining companies.

During 2003, in furtherance of its economic policy, the Ministry commenced the 5 year Projet de Gouvernances des Ressources Minérales program, with the following objectives:

To further improve and enforce the legal and statutory framework, particularly with respect to mining.

To promote investment in the minerals sector through the dedicated Agence de Promotion du Secteur Minier.

To improve the geoscientific knowledge of Madagascar through geophysical surveys, geological mapping, and remote sensing, with appropriate staff training to support mapping Projects.

To address environmental health and safety issues and to contribute to poverty reduction.

4.3 Location

The Project is located approximately 160 km south-east of Madagascar's port city of Toliara, in the Tulear region and roughly 220 km north-west of Fort Dauphin. Refer (Figure 2) below.

Figure 2: Molo Project Location

4.4 Property Area

The Project includes 790 claims and an area totalling 308.6 km^2. Refer Figure 3 below. The Project is centred on UTM coordinates 495,289 easting 7,345,473 northing (UTM 38S, WGS 84 datum). The Project is located 11.5 km east-north-east of the town of Fotadrevo within Exploitation / Mining Permit PE #39807 which covers an area of 175 km^2 or 17,500 hectares (“ha”) and Exploration Permits PR #39806 and PR #39810 which cover areas of 96.1 km^2 (9,609 ha) and 37.5 km^2 (3,750 ha) respectively. The Government of Madagascar designates individual claims by a central LaBorde UTM location point, comprising a square measuring 625m x 625m.

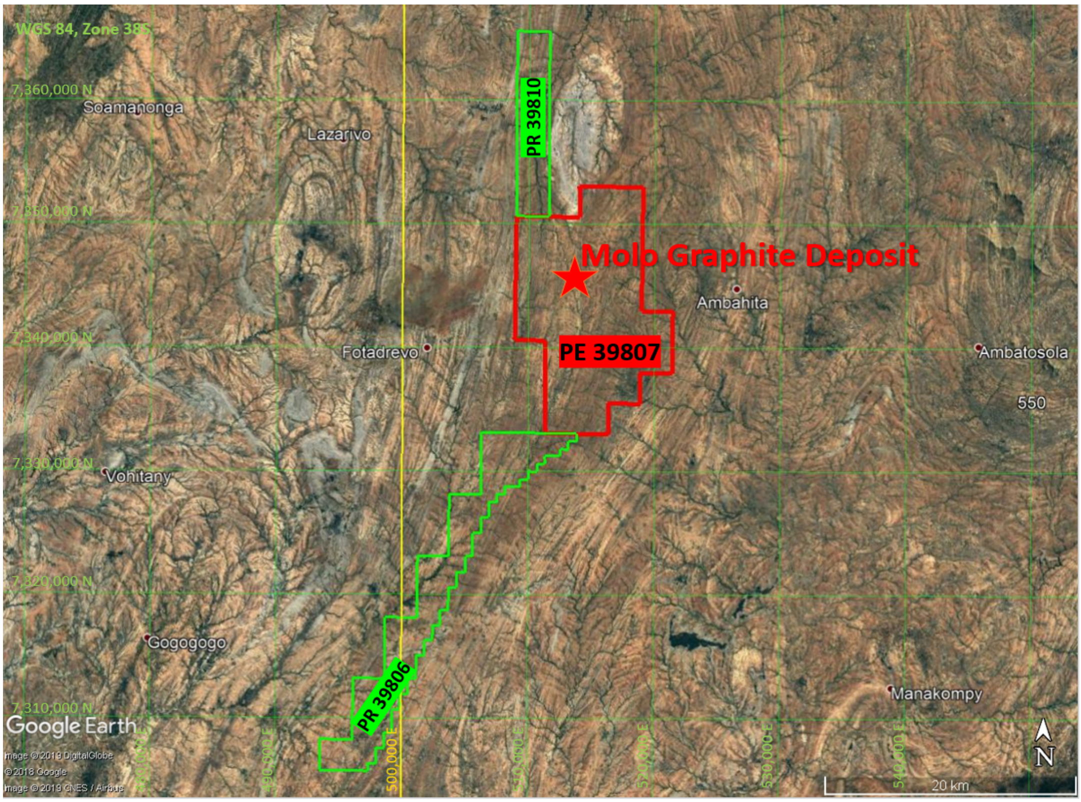


Figure 3: Molo Project - Exploitation Permit PE #39807

4.5 Mineral Tenure and Title

On December 14, 2011, the Company entered into a Definitive Joint Venture Agreement ("JVA") with Malagasy Minerals Limited ("Malagasy"), a public company on the Australian Stock Exchange, to acquire a 75% interest to explore and develop a group of industrial minerals, including graphite, vanadium and approximately 25 other minerals. On October 24, 2013, the Company signed a Memorandum of Understanding ("MOU") with Malagasy to acquire the remaining 25% interest in the land position.

On April 16, 2014, the Company signed a Sale and Purchase Agreement and a Mineral Rights Agreement with Malagasy to acquire the remaining 25% interest. Malagasy retains a 1.5% net smelter return royalty ("NSR").

CCIC reviewed a copy of the Contrat d'Amodiation pertaining to this right and are satisfied that the rights to explore this permit have been ceded to the Company, or one of its Madagascar subsidiaries.

The Project was located within Exploration Permit PR #3432 as issued by the BCMM pursuant to the Mining Code 1999 (as amended) and its implementing decrees. On January 18, 2019, Permit PR #3432 was transformed into two Exploration Permits (PR #39806 and PR #39810) and an Exploitation Permit (PE #39807) by the Ministry of Mines, with the official permit being granted to the Company by the BCMM on February 14, 2019.

Mineral Resources and Reserves delineated in Sections 14 and 15 of this report are entirely within the bounds of Exploitation Permit PE #39807. The Company holds the exclusive right to exploit / mine and explore for graphite within this license area for a period of 40 years and can renew the license several times for a period of 20 years upon each renewal.

The Company holds the exclusive right to explore for a defined group of industrial minerals within Exploration Permits PR #39806 and PR #39810. These industrial minerals include the following: Vanadium, Lithium, Aggregates, Alunite, Barite, Bentonite, Vermiculite, Carbonatites, Corundum, Dimensional stone, (excluding labradorite), Feldspar (excluding labradorite), Fluorspar, Granite, Graphite, Gypsum, Kaolin, Kyanite, Limestone / Dolomite, Marble, Mica, Olivine, Perlite, Phosphate, Potash–Potassium minerals, Pumice Quartz, Staurolite, Zeolites.

Companies in Madagascar first apply for an exploration mining permit with the BCMM, a government agency falling under the authority of the Minister of Mines. Permits under usual circumstances are generally issued within a month. The number of squares varies widely by claim number.

The updated Decret requires the payment of annual administration fees of Permits Research of 15,000 Ariary (MGA) for exploitation permits in years one and two. Annual fees increase by multiplying by a factor equivalent to the number of years (plus 1) that the company has held the permit. Exploration permits have an updated duration of 5 years, with the possibility of two renewals of an additional 3 years each. Payments of the administration fees are due each year on 31 of March of each year, along with the submission of an activity report. Each year the Company is required to pay a similar, although increasing, amount in order to maintain the claims in good standing.

Reporting requirements of exploration activities carried out by the title holder on an Exploration Permit are minimal. A title holder must maintain a diary of events and record the names and dates present of persons active on the Project. In addition, a site plan with a scale between 1/100 and 1/10,000 showing “a map of the work completed” must be presented. CCIC is of the opinion that the Company is compliant in terms of its commitments under these reporting requirements.

The Project has not been legally surveyed; however, since all claim boundaries conform to the pre-determined rectilinear LaBorde Projection grid, these can be readily located on the ground by use of Global Positioning System (“GPS”) instruments. Most current GPS units and software packages do not, however, offer LaBorde among their available options, and therefore defined shifts must be employed to display LaBorde data in the WGS 84 system. For convenience, all the Company’s positional data is collected in WGS 84 and if necessary, converted back to LaBorde.

4.6 Royalties

The Madagascar government retains a 2% gross revenue royalty, Vision Blue retains a 3% gross revenue royalty, and Malagasy retains a 1.5% net smelter return royalty on the Project.

4.7 Permits

Exploitation Permit PE #39807 (175 km^2) and Exploration Permits PR #39806 and PR #39810 are held under the name of a subsidiary of the Company called ERG (Madagascar) Ltd. S.A.R.L.U. and were granted to the Company by the BCMM on February 14, 2019.

The Madagascar Ministry of Environment's Office National pour l'Environnement (the National Office for the Environment), or "ONE", granted the Company its Environmental License for Phase 1 of the Molo Graphite Mine on April 8, 2019 after reviewing the following:

- Exploitation Permit PE #39807.
- Environmental and Social Impact Assessment ("ESIA") and RAP to International Finance Corporation (IFC) Performance and World Bank Standards.
- Completion of local and regional stakeholder and community engagement, with overwhelming support from both the local community and local government, as well as regional government.
- Signed agreements with all potentially affected land occupants to accept compensation for any affected crops and grazing land and relocation if needed.
- Approved capital investment certification from the BCMM.
- Receipt of Cahier des Charges Minière (mining specification) from the BCMM as pre-requisite to submitting the ESIA and RAP to ONE for review
- Successful completion of the ONE's technical evaluation process which consisted of a site visit and four separate community consultations.
- Joint agreement and signature of the Cahier des Charges Environnementales (environmental specification) with the ONE

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Access

Access to Molo from Toliara, is initially via a 70 km paved road to the village of Andranovory (Figure 4) below. From Andranovory, secondary all season roads continue to Betioky, distance of 93 km. From Betioky the Property area can be reached via Ambatry to Fotadrevo, a distance of 105 km (268 km total), or from Betioky to Ejeda, then onwards to Fotadrevo, a distance of 161 km (324 km total). This alternate route from Ejeda to Fotadrevo is used by heavy transports and by all vehicles during portions of the rainy season, as the primary route quickly becomes impassable. At the height of the rainy season, both routes to Fotadrevo may be largely impassable. Molo may be reached from Fotadrevo by a well-maintained dirt road.

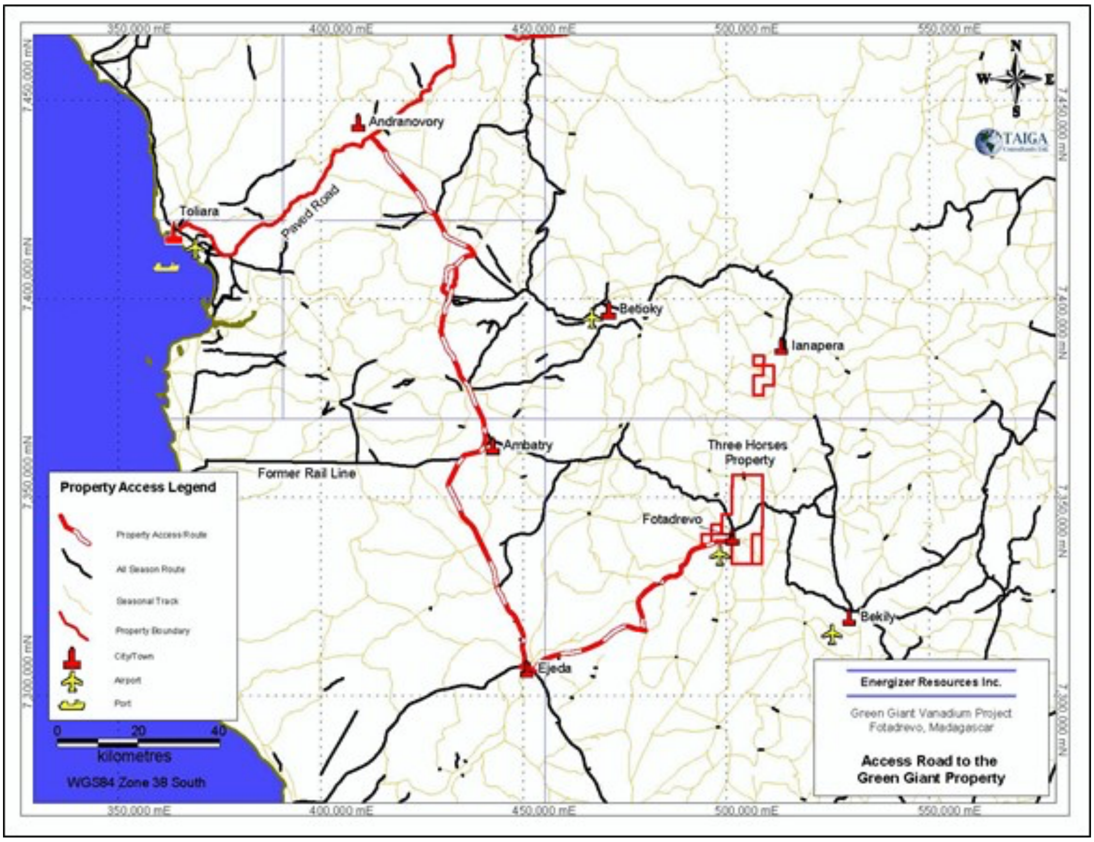


Figure 4: Road Access to the Molo Area from the Town of Toliara

With the upgrading of an existing airstrip at Fotadrevo (Figure 5) to an all-weather airstrip during the 2008 exploration programme, the Project area is now accessible year round, (except under special circumstance caused by continuous or multiple days of heavy rain) by air, using private aircraft out of Antananarivo. Flying times to Fotadrevo are roughly 2.5 hours from Antananarivo and 45 minutes from Toliara.

Figure 5: Airstrip At Fotadrevo

Antananarivo is currently serviced by Air France (from Paris) and Air Mauritius (Sir Seewoosagur Ramgoolam International Airport in southeast Mauritius). Madagascar due to current COVID constraints only allow repatriation and cargo flights from South Africa and Mauritius. Air Madagascar also has regularly scheduled domestic jet and propjet flights throughout the country, including daily flights between Antananarivo and Toliara.

5.2 Physiography

The Molo deposit area is covered by sparse vegetation (Figure 6) below. Grass cover is widespread and trees are widely spaced overall, with accumulations near drainage lines and streambeds. In areas of lower relief, alluvial cover is generally shallow, and bedrock and/or float are readily observable. Elevations range between 536m to 565m above mean sea level ("mamsl").

Figure 6: View of the Molo Project Area Showing the Nature of the Vegetation

Typical of the tropics, the surface is subject to lateritic weathering. However, full laterite profiles are rarely developed within the southern climatic zone. Previous drilling on the Property indicated that the weathered profile is typically less than 10m thick in the region, which is roughly one third of that seen in other parts of Madagascar and on the adjacent African continent.

5.3 Climate

Five climatic zones divide Madagascar. The Molo deposit area falls within the semi-desert south zone, with elevated temperatures year-round peaking in the hot season at an average of over 30°C. The climate is dominated by south-eastern trade winds originating in the Indian Ocean anticyclone, a centre of high atmospheric pressure that seasonally changes its position over the ocean. Madagascar has two seasons, a hot, rainy season from December to March / April, and a cooler dry season from April / May to November. Total rainfall is sparse within the Molo area, with yearly precipitation ranging from 30 cm to 50 cm. The rainy season causes difficulty in travelling off the main highways and for exploration, effectively limiting drilling to the dry season.

5.4 Local Resources and Infrastructure

The village of Fotadrevo, (where the Company has its base camp), is located to the west of the Molo deposit area. The village has been a labour source during the exploration programmes on Molo and will most likely provide a portion of the workforce for Molo Phase 1 and subsequently modular Phase 2. A few basic goods are commercially available in the village. However, the main centre for support of exploration and development are the cities of Toliara and Antananarivo. Two 40 kVA diesel powered generators provide power to the Company's base camp.

A cellular telephone tower is located in Fotadrevo, which provides convenient coverage. Currently there is an existing well on the Molo property with a further 2 to be drilled for Phase 1. The expected requirement for Phase 2 is a further 22 holes to be drilled.

5.5 Security

As Madagascar is an island no border issues, or conflicts are known that might affect operations, security, or title in the region. Security of personnel is a company policy directed by management. Considering that the area is predominantly rural, few police, or other security patrols are common in the area. There is always a small possibility that local criminal activity might affect operations, and to mitigate this, the company employs the local military forces to accompany field parties away from secure areas. The Madagascar government provides a requested number of regular military troops, at a cost to the Company, to ensure security on the Property, on the work site and for the company's equipment.

6 HISTORY

The region around the Property has primarily been explored for base metal type occurrences, although colonial geologic services were alert to all kinds of mineral potential in the region. In 1985 the Bureau de Recherches Géologiques et Minières ("BRGM") (http://www.brgm.fr/) produced a three-volume country scale compilation of all exploration and mineral inventory data in their files. Relatively little exploration and development work has been completed in south-western Madagascar after that of BRGM, and therefore these volumes are key to retracing any historical data. Archival research by the Company has not revealed evidence of mineral exploration in the past fifty years within the Project area prior to the exploration work completed by the Company.

6.1 Property-Scale Exploration History

Prior to the exploration work completed by the Company, (then Uranium Star) in 2007 there is no record of any previous exploration activity within the Project area and no historical resource estimates exist for the area, or for Molo. Between 2007 and 2011 the Company retained Taiga Consultants Limited ("Taiga") to manage exploration activities on the Project. Table 10 shows a summary of the historical exploration activities previously on the property. The PEA is informed by all previous exploration work execution of Phase 1 of the Molo Graphite Project and no further exploration work was completed for the PEA.

Table 10: Historical Activities on the Project

Date	Activity	Company Responsible
2011	Prospecting (538 grab samples) over areas of historical graphitic occurrences (BRGM) on the Project	The Company
	Diamond Drilling (7 holes). Metallurgical samples selected from Molo-11-07	Boart Longyear
	Trenching programme (1 trench)	
	Geologic mapping at 1:1000 scale	The Company
	EM-31 ground geophysical survey (52.2 line kms)	The Company

Exploration work undertaken in 2011 led to the discovery of the Molo deposit, which then became the focus of the 2012-2014 exploration programmes addressed in the FS and this PEA.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

Madagascar comprises a fragment of the African Plate, which rifted from the vicinity of Tanzania at the time of the breakup of Gondwana, some 180 million years ago. At that time Madagascar remained joined with India, moving east-by-south until the late Cretaceous (approximately 70 million years ago), whereupon the two land masses split apart. On a regional scale Madagascar can be described as being formed by two geological entities, a Precambrian crystalline basement, and a much younger Phanerozoic sedimentary cover (Figure 7) below that hosts potentially economic coal deposits. The central and eastern two-thirds of the island are mainly composed of Neoproterozoic-aged, crystalline basement rocks, composed of a complex mélange of metamorphic schist and gneiss intruded by younger granitic and basic igneous rocks. The Phanerozoic sedimentary cover is largely restricted to the western side of the island and is Carboniferous to Permian-Triassic. These rocks correlate with the Karoo Super group successions of sub-Saharan Africa, which was widespread in the former super continent of Gondwana.

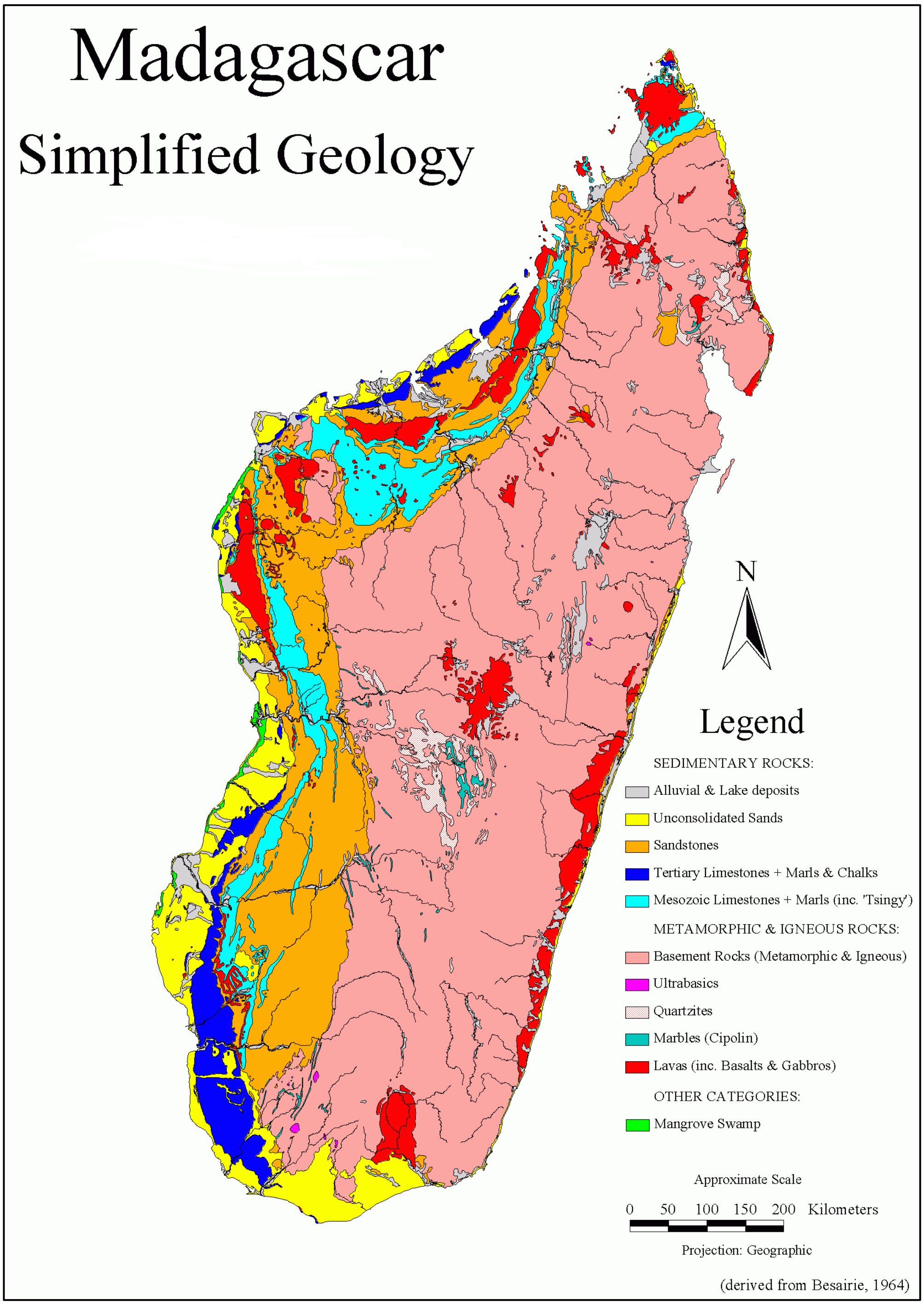


Figure 7: Geological Map of Madagascar Showing the Distinctive Crystalline Basement and Sedimentary Basins in the West (Source: Besarie (1964)

The geology of the basement of Madagascar is composed of inter-continental tectonic blocks made up of ancient poly-deformed, high-grade metamorphic rocks and later igneous intrusions. The tectonic and metallogenic basement framework was originally subdivided into four blocks (Besarie, 1967), these being the: northern Bemarivo Block; north-eastern Antongil Block; central Antananarivo Block; and the southern Bekily Block. The Molo deposit lies entirely within the bounds of the Bekily Block (Figure 8) below. Later authors (e.g. Pitfield et al., 2006) divided the Precambrian basement of Madagascar in a somewhat different manner, with nine tectono-metamorphic units (Figure 9). In the case of the region around Molo, the tectonic blocks and the tectono-metamorphic units cover a nearly identical area, and as such these divisions can be used interchangeably.

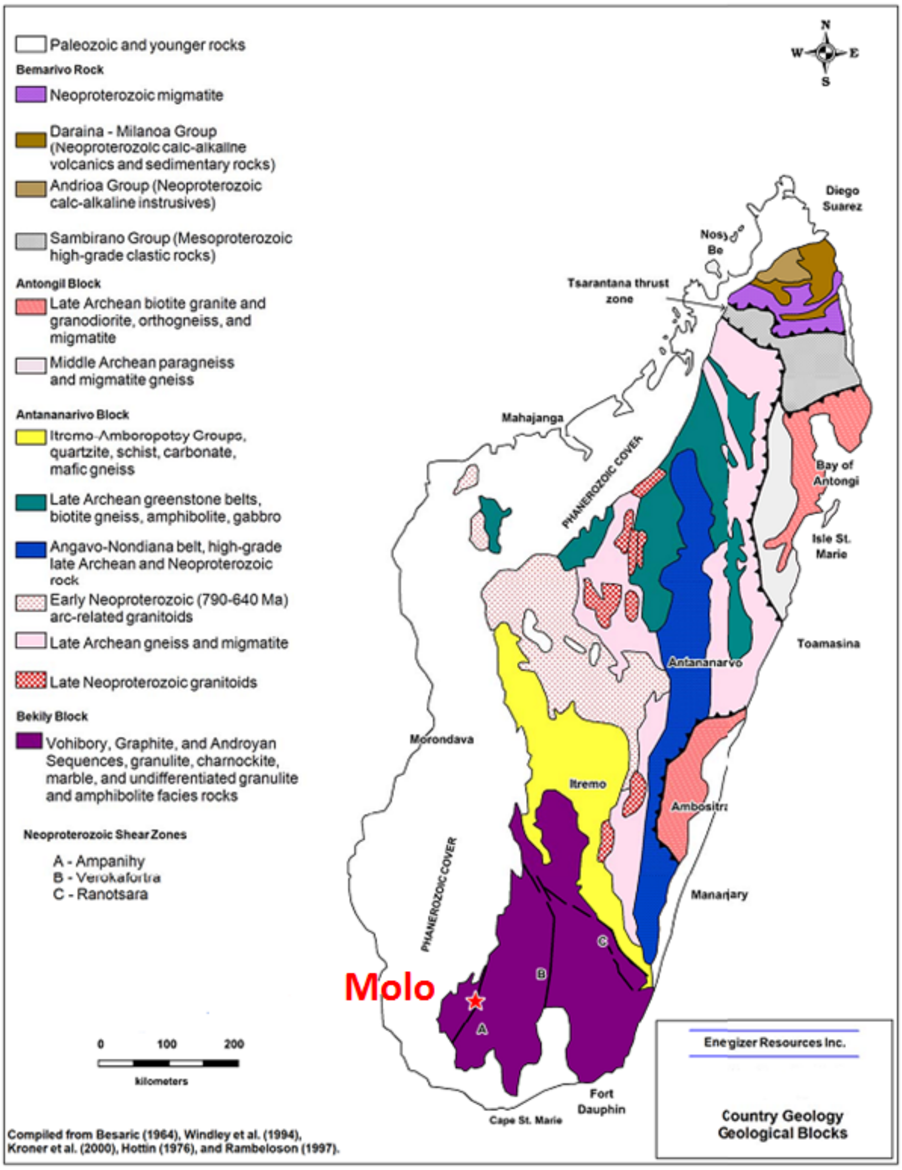


Figure 8: Country Geology: Geological Blocks (source: AGP Mining Consultants (2011))

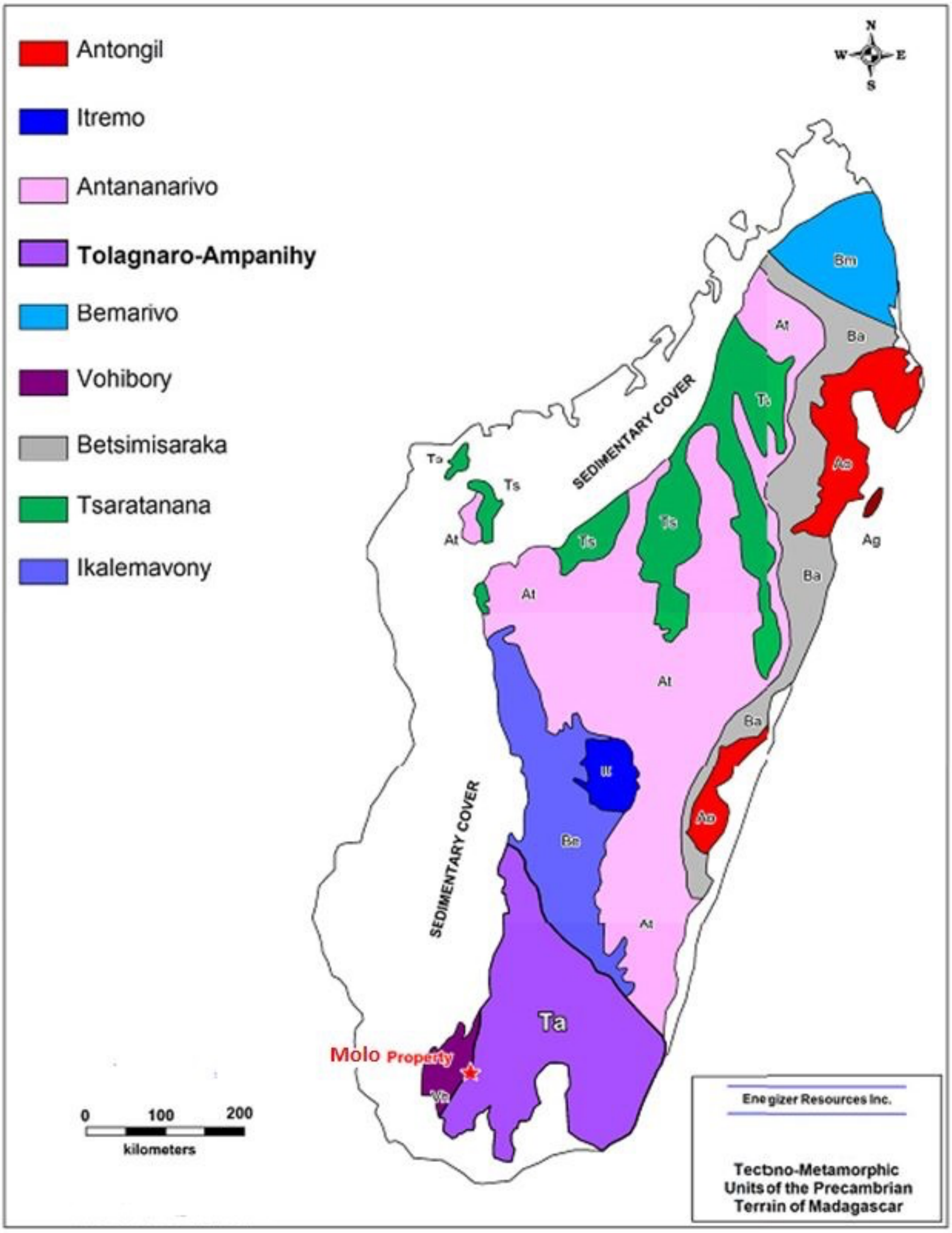


Figure 9: Tectono-Metamorphic Units of the Precambrian Terrain of Madagascar (source: AGP Mining Consultants, 2011)

7.1.1 Tectonic History of Southern Madagascar

Southern Madagascar forms part of the Mozambique Mobile Belt and is made up of a section of lower Proterozoic crust that underwent granulite-facies metamorphism during the Pan-African Orogeny (Paquette et al., 1994). Three crustal units separated by north-south trending vertical shear zones make up this area (Figure 10). Each of these units experienced granulite facies metamorphism with temperatures between 750˚C and 800˚C.

The associated pressures during the Pan-African Orogeny in this area range between 3 to 11 kilobars ("kbar") with a decreasing trend from west to east (Pili et al., 1997).

The Bekily block, (also referred to as the Androyen region, or the Tolagnaro-Ampanihy tectono-metamorphic unit), forms a vast high-grade meta-sedimentary (paragneiss) terrane that has been metamorphosed to granulite facies conditions. This region comprises a complex Neoproterozoic terrain of high-grade metamorphic rocks, with a history of polyphase deformation. Two prominent north-south trending late Neoproterozoic ductile shear zones, the Ampanihy and Vorokafotra Shear zones, cross-cut the region. A third set of en-echelon shears forms part of the early Palaeozoic Ranotsara Shear Zone that cuts the basement in a north-west-south-east direction over a strike length of over 400 kms.

De Wit et al. (2001) recognize four episodes of deformation and metamorphism. The two early episodes of simple shear deformation (D1 and D1), during which north-east verging recumbent sheath folds and ductile thrusts were formed, are dated between 647 Ma -627 Ma. Early prolate mineral fabrics (L1/L2) are preserved in massif type anorthosite bodies and their marginal country rocks.

D1 and D2 deformation was followed by a 10 Ma to 15 Ma period of static, annealing metamorphism when bulk shortening (D3) took place. D2 and D3 deformations are coaxial but are separated in time by leucocratic dykes that intruded between 620 Ma and 610 Ma. Between 609 Ma and 607 Ma, D3 deformation was focused zonally, forming the prominent north–south shear zones. Oblate strain resulted in a strong composite D2/D3 fabric defined by sub-vertical S-tectonites and sub-horizontal intersection lineations.

A variety of post-D3 pegmatites accompanied the following 85 million years of relatively static annealing and metasomatic / metamorphic mineral growth. Numerous occurrences of phlogopite, uranium, and rare earth elements are associated with these pegmatitic bodies. A continuum of concordant monazite dates of between 605 Ma and 520 Ma suggests that this thermal event is part of an extended period of low-pressure (3k to 5k bar) charnockite producing event.

The D4 deformational event recorded within the Ranotsara Shear zone overlaps with the youngest parts of the regional metamorphic conditions. Between 530 Ma to 490 Ma, prevailing low pressure, high temperature amphibolite-granulite facies rapidly gave way to greenschist facies conditions.

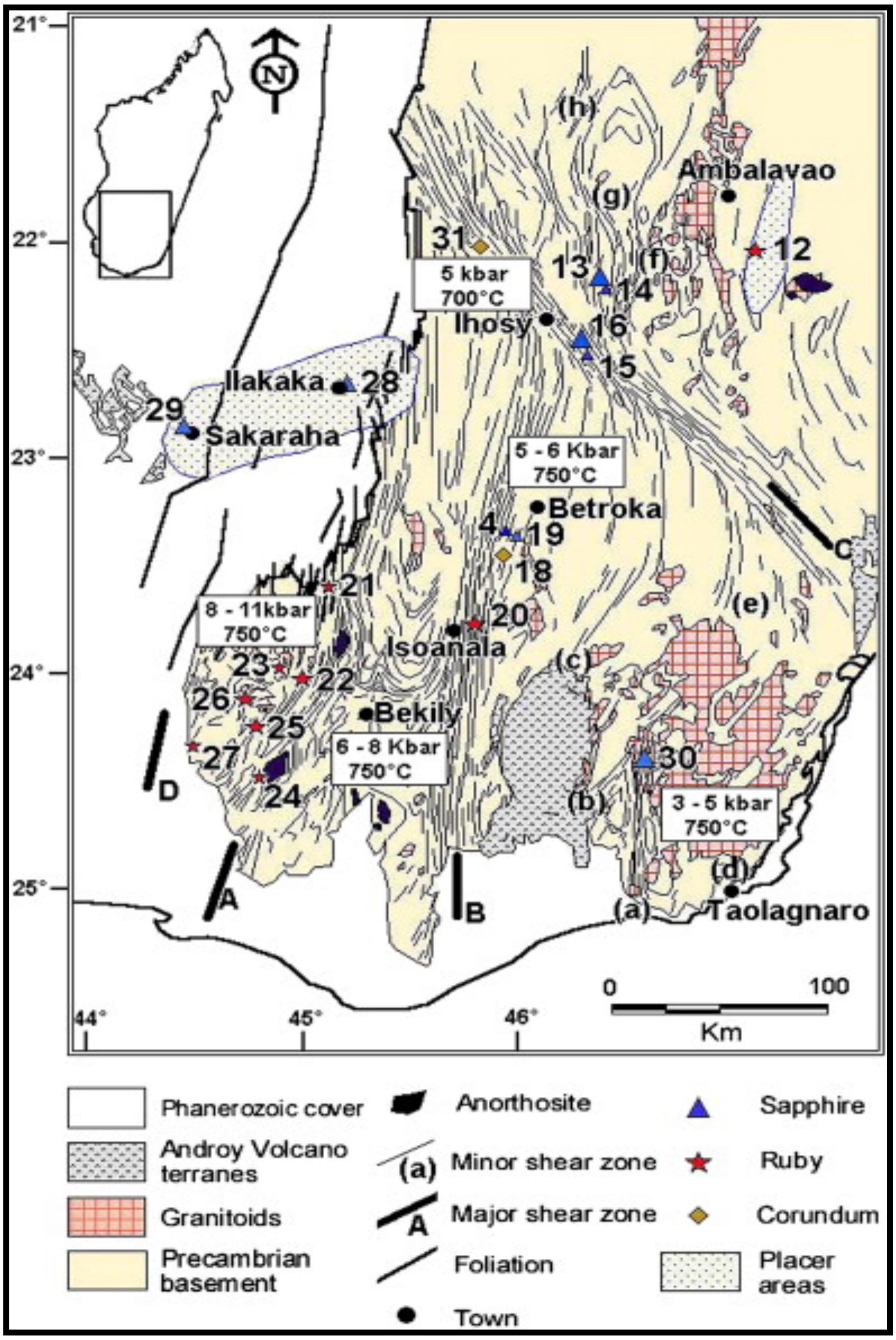
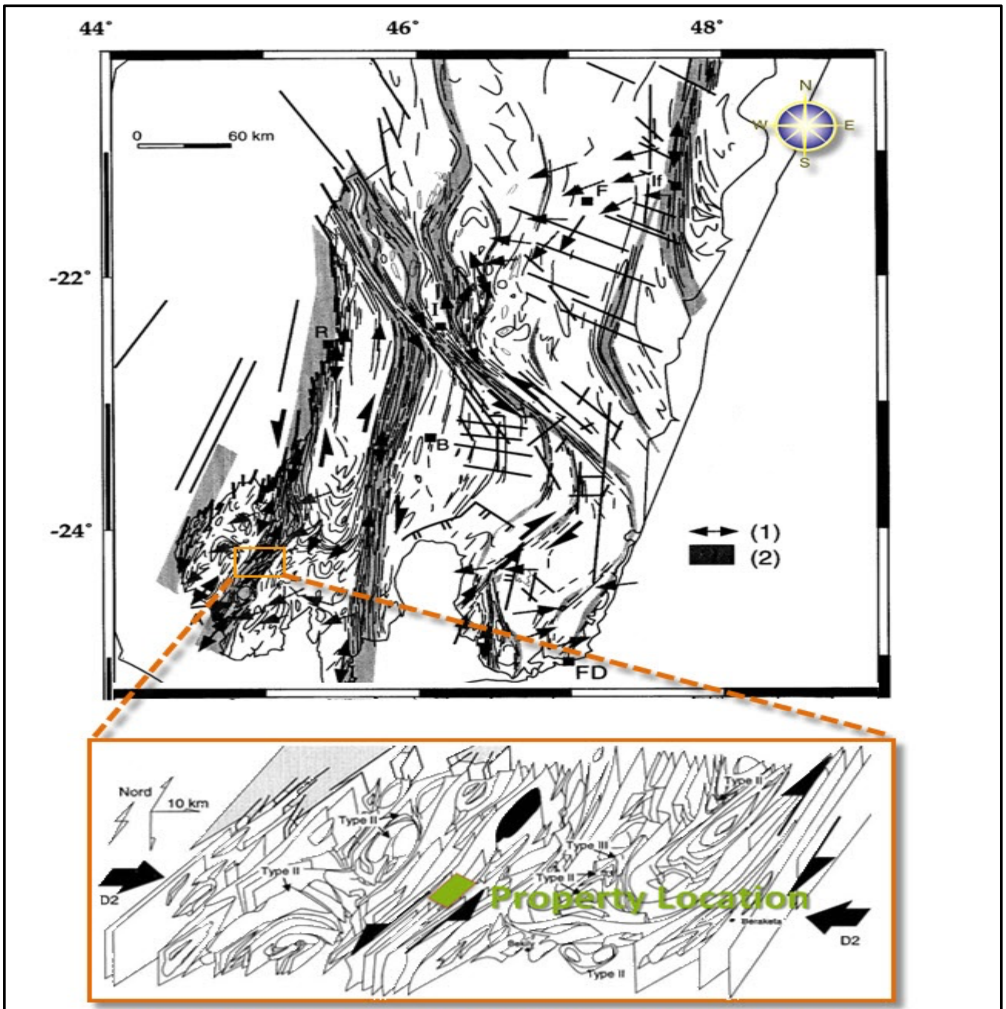


Figure 10: Structural and Lithological Sketch Map of South-East Madagascar Showing the Positions of yhe Major Shear Zones (source: Rakotondrazafy et al.,2008)

7.2 Regional Geology as it Relates to the Molo Deposit

The Molo deposit occurs within the regional Ampanihy Shear Zone (Figure 11). The most conspicuous feature of rocks found within this shear zone is their well-developed north-south foliation and vertical to sub-vertical nature. Martelat et al. (2000) state that this observed bulk strain pattern is clearly related to a transpressional regime during bulk horizontal shortening of heated crust, which resulted in the exhumation of lower crustal material. Figure 11 below illustrates the general position of Molo relative to the D2 regional strain pattern and the resulting Ampanihy Shear zone.

The Project area is underlain by supracrustal and plutonic rocks of late Neoproterozoic age that were metamorphosed under upper amphibolite facies and deformed with upright north-northeast-trending structures. The supracrustal rocks involve migmatitic (± biotite, garnet) quartzo-feldspathic gneiss, marble, chert, quartzite, and amphibolite gneiss. The metaplutonic rocks include migmatitic (± hornblende / diopside, biotite, garnet) feldspathic gneiss of monzodioritic to syenitic composition, biotite granodiorite, and leucogranite.



Finite strain pattern of southern Madagascar with Ampanihy transpressive lithospheric shear zone shown in inset image

Modified from Martela, 2000

Figure 11: Position of the Molo Project Area within the Overall Strain Pattern Documented for Southern Madagascar

Descriptions of the individual lithological units identified by the Company, which are relevant to Molo, are included below.

7.2.1 Lithological Descriptions of Individual Rock Formations

7.2.1.1 Amphibolitic Gneiss

Dark grey to black, mesocratic to melanocratic, medium to coarse grained, sub-equigranular to porphyroblastic amphibolitic gneiss and amphibolite. Amphibolitic gneiss forms one, or more major continuous bands in the eastern part of the permit, intercalated with quartzo-feldspathic gneiss and spatially associated with marble. In the central portion of the detailed map area, amphibolitic gneiss forms local bands, or lenses intercalated with quartzo-feldspathic gneiss and marble.

7.2.1.2 Meta-quartzite

White to greyish white, weakly to moderately layered and foliated, coarse to medium grained quartzite. Brecciated quartzite with isoclinally folded layering is locally associated with dark brown ferruginous gossan. Un-brecciated quartzite very locally contains narrow, concordant, and discontinuous seams of gossan.

7.2.1.3 Grey-white Chert

Mottled greyish-white, massive to brecciated, hyalocrystalline graphite-bearing chert, (or possibly siliceous rhyodacite). Grey-white chert displays evidence of polyphase brecciation, involving cm to mm scale, angular white siliceous fragments in a relatively early translucent grey siliceous (chalcedony) breccia matrix, and/or a later opaque brown ferruginous gossan breccia matrix.

7.2.1.4 Brown Fe-carbonate Chert

Tawny (yellowish) brown to reddish brown and chocolate brown, massive, hyalocrystalline opaque, graphite-bearing Fe-carbonate chert, variable biotite, and/or specularite. Brown chert, like grey-white chert, contains a small amount (≤1%) of fine-grained disseminated graphite, as well as variably small amounts of fine-grained disseminated biotite and/or specularite. Brown chert represents a widespread Fe-carbonatized alteration facies of grey-white chert, and both occur within the same chert masses. Brown chert is intimately associated with brown marble and ferruginous gossan.

7.2.1.5 Ferruginous Gossan

Dark purplish brown to black, dense, massive to brecciform and quasi-layered, aphanitic to fine-grained, siliceous ferruginous gossan. The gossan is variably highly siliceous to moderately siliceous and pitted, composed in part of Fe carbonate (siderite-ankerite) and generally contains disseminated to clustered, fine-grained specularite, biotite, and/or graphite. Siliceous ferruginous gossan occurs as:

Breccia matrix of late-stage chert breccia and quartzite breccias.

Concordant layers inter-calated with chert and marble and discontinuous concordant seams in quartzite discordant masses cutting regional structure in quartzo-feldspathic gneiss and marble.

Siliceous ferruginous gossan is locally associated with cm scale patchy masses of green, opaque calc-silicate, or bright green amorphous and resinous calc-silicate mineral.

7.2.1.6 Quartz Feldspar Gneiss

Light grey to white, migmatitic, well foliated, and locally lineated, leucocratic to hololeucocratic, generally medium-grained (to fine, or coarse grained), ubequigranular to porphyroblastic biotite-garnet Quartzo-feldspatic gneiss comprises a mixture of fundamental constituent lithologies, dependent on the relative abundance, or absence of biotite and garnet.

7.2.1.7 Feldspathic Gneiss

Pinkish grey to pink, migmatitic, foliated, medium to coarse grained, leucocratic (± hornblende/ diopside, biotite, garnet) feldspathic gneiss. The feldspathic gneiss is comprised of a mixture of quartz-poor constituent lithologies.

7.2.2 Structural Geology

In 2010 a structural interpretation was undertaken based on the 2007 Fugro Airborne Surveys (http://www.fugroairborne.co.za/) ("Fugro") helicopter-borne frequency domain electromagnetic (DIGHEM V) multi-coil, multi-frequency, electromagnetic and high sensitivity cesium magnetometer geophysical survey (Butler, 2010; in Desautels et al., 2011). Only magnetite bearing units were capable of being interpreted, except where putative intrusions cross-cut the main fabric in a magnetised area. This work showed the Green Giant Property to be dominated by structures associated with the Ampanihy Shear Zone. Butler (2010; in Desautels et al., 2011) specifically identified three magnetic domains associated with the Ampanihy shear system:

Zones where magnetic units are parallel, or near parallel to the walls of the domain. In these regions, the shearing has reduced intrafolial folds into sheared out 'tectonic fish'. There are also broad zones where a low content of magnetite (± pyrrhotite) may be present which most likely represent a different metamorphic mineral assemblage, (different pressure and temperature conditions), or pre-metamorphic alteration and/or rock types.

Zones where magnetic units vary from parallel to a high angle at the domain boundary. These regions are interpreted to be the intrafolial fold remnants of sheath folds. The boundary shear of these domains may represent a sheared-out early thrust, or high angle fault.

Zones with refolded chaos folds in domain lozenges. These regions occur in the north-central portion of the study area and may be a remnant of a broad F3 episode enclosed within intense zones of ductility.

7.3 Molo Property Geology

The Molo graphitic zone consists of multi-folded graphitic strata with a surficially exposed strike length of over two kilometres (Figure 12). Outcrop mapping and trenching on Molo has shown the surface geology to be dominated by resistant ridges of graphitic schist (Figure 13) and graphitic gneiss, with fracture-lined vanadium mineralisation, as well as abundant graphitic schist float.

Geological modelling (Figure 14) has shown that the deposit consists of various zones of mineralised graphitic gneiss, with a barren footwall composed of garnetiferous gneiss (Figure 15). The host rock of the mineralised zones is graphitic gneiss.

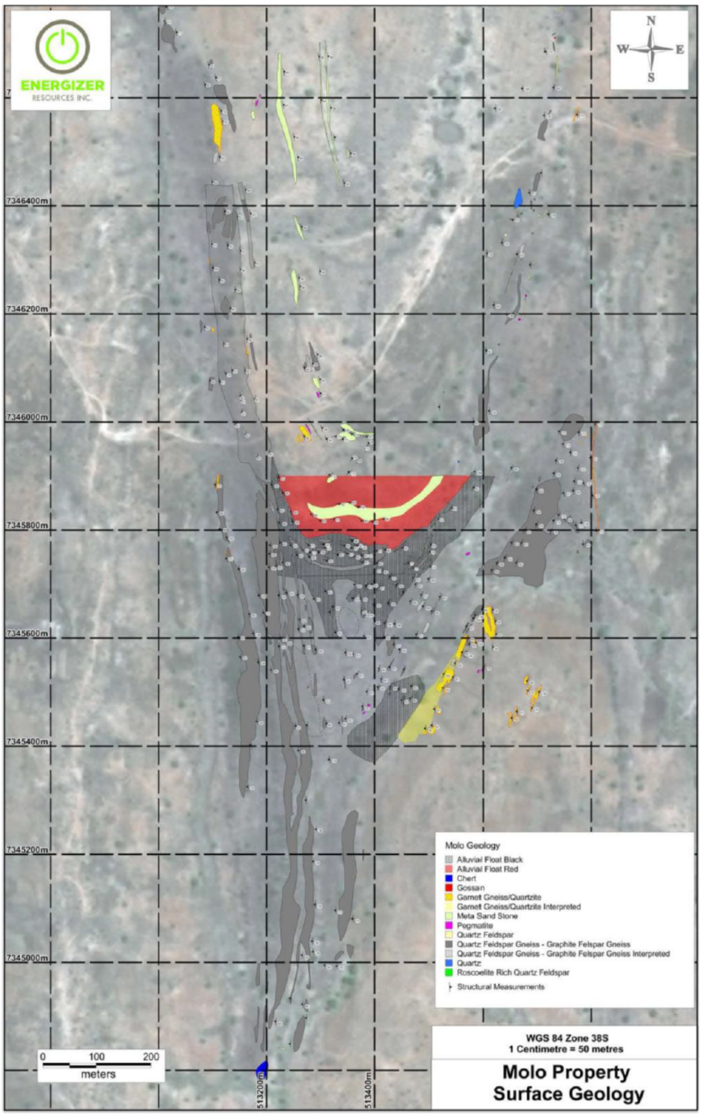


Figure 12: Map Showing the Surface Geology of the Molo Deposit

Figure 13: Outcrop Exposure of Graphitic Schist on the Molo Project

Photograph taken during the site visit of the May 10, 2012

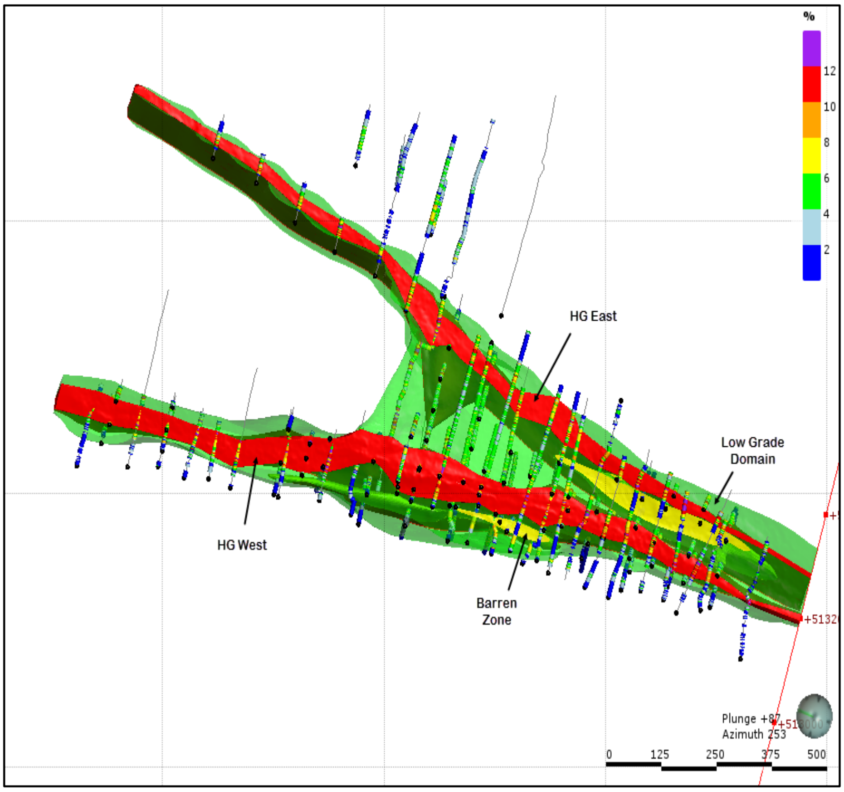


Figure 14: Geological Model of the Molo Deposit Showing the Nature of the Two Tightly Oppressed 'High" Grade (Hg East and Hg West) Mineralized Zones

Figure 15: Garnet Gneiss in the Footwall of Molo-4 at a Depth of 132m

No academic studies have been undertaken on the graphitic schists and gneisses of the Molo deposit and at present the deposit is still not fully understood. There is, however, no indication of secondary hydrothermal, or other transported, post-metamorphic graphitic mineralisation, or upgrading and the present distribution and crystallinity of the graphite zones seem to be primarily due to regional metamorphic and structural events. The length, width, depth and continuity of mineralisation for the Molo deposit are more fully described under Section 14 “Mineral Resource Estimates”.

8 DEPOSIT TYPES

The Green Giant-JV Project hosts at least two different deposit types:

Metamorphosed black shale / roll front redox vanadium deposits, and deposit types.

Flake graphite deposits. The metamorphosed black shale / roll front redox vanadium deposits are well described in previous reports (e.g. Scherba and Chisholm, 2008) and are not repeated here.

8.1 Graphite Deposit Types

Graphite is one of the three familiar naturally occurring forms of the chemical element Carbon ("C"). The other two varieties are amorphous carbon, (not to be confused with amorphous graphite) and diamond. Graphite may be synthetically produced, or from natural source. Most natural sources are considered one of three main types, amorphous, flake, or vein.

The following is taken mainly from Kogel et al. (2006):

Graphite is widely distributed throughout the world, occurring in many types of igneous, sedimentary, and metamorphic rocks.

Many occurrences, however, are of little economic importance.

The more important occurrences are those found in metasomatic-hydrothermal deposits and in sedimentary rocks that have been subjected to regional, or contact metamorphism

Economic deposits of graphite include the following:

- Flake graphite disseminated in metamorphosed, silica-rich.
- Sedimentary rock.
- Flake graphite disseminated in marble.
- Amorphous deposits formed by metamorphism of coal.
- Carbon-rich sediments.
- Veins filling fractures, fissures, and cavities in country rock.
- Contact metasomatic or hydrothermal deposits in metamorphosed.

Natural graphite of economic value can be divided into two main classes, these being:

- Disseminated flake.
- Crystalline vein (fibrous, or columnar).

Most, if not all, of the world's deposits of flake graphite occur in metamorphic rocks of Precambrian age. Flake graphite is a lamellar form found in metamorphic rocks, such a marble gneiss, and schist. Each flake is separate, having crystallized as such in the rock. In many cases, pegmatitic veins have intruded the rocks.

Crystalline vein graphite (also called lump, or high crystalline graphite) is normally found in well-defined veins, or pocket accumulations along intrusive contacts of pegmatites with limestones. Here the enclosing wall rock is not necessarily graphitic. This type of deposit assumes the character of a true lode. The graphite in these deposits is of two types, foliated and columnar. The Sri Lankan graphite deposits are of vein type.

8.2 Graphite Mineralisation on Molo

Petrographic descriptions undertaken on thin sections of selected rocks of the Manga vanadium deposit submitted for metallurgical analysis to Mintek (www.mintek.co.za/) in 2010 identified 17.17% modal graphite from the silicate composite, and 15.87% modal graphite from the oxide composite samples. Three additional composite samples were submitted to Mintek at the conclusion of the 2010 exploration programme. The Quantitative Evaluation of Minerals by Scanning Electron Microscopy ("QEMSCAN") analysis of these samples quantified a graphite composition of 4.09%, while the head chemical analysis quantified a graphitic carbon content of 3.87%.

The identification of graphite as a potential credit to the Company's vanadium resources led the Company's geologists to conduct a reconnaissance exploration programme with the goal of delineating new graphitic trends and comparing them to those associated with vanadium mineralisation. During the period of this reconnaissance exploration programme various surficial graphitic trends were identified on the Green Giant Property. These graphite trends were visually determined to be of both higher carbon content, and larger flake size than those associated with the vanadium resource mineralisation. Samples from these mineralised zones were submitted to Mintekhttp://www.mintek.co.za/ for analysis, as well as to the North Carolina State University ("NCSU") Minerals Research Laboratory in Asheville, North Carolina.

9 EXPLORATION

The identification of graphite as a potential credit to the Company's NI 43-101 compliant vanadium resources (Scherba and Chisholm, 2008) led to a reconnaissance exploration programme being undertaken on the Property in September 2011, with the goal of delineating new graphitic trends. Activities during this phase of exploration included prospecting, grab and trench sampling, and diamond drilling. Based on the results of this programme, the Company launched a second phase of exploration in November 2011. The objective of this second programme was to use geophysical techniques to delineate additional graphite mineralisation, as well as to drill test the known graphitic.

The signing of the JV agreement with Malagasy in November 2011 prompted additional exploration to ascertain the industrial mineral potential of the JV Property area. Exploration activities consisted of, geologic mapping, prospecting and sampling, (including metallurgical), ground geophysical surveying (EM-31), trenching, and diamond drilling. As a consequence of work undertaken during 2011, the Molo graphite prospect was identified and targeted for additional work, which was undertaken between May 2012 and June 2014. As a result of the preceding Molo Phase 1 is currently in execution and will be immediately adjacent to the planned Molo modular Phase 2 plant.

9.1 Geological Mapping

A series of excellent 1/100,000 scale geological maps (1952-53) are available for the region surrounding Molo (Fotadrevo-Bekily, Ianapera, Sakamena-Sakoa), with the area covered by the 1/100,000 scale topographic map #H-60 Fotadrevo.

Various mapping Projects have been completed on the entire Green Giant Property, with the major emphasis being strictly dedicated to the commodity of interest at that time. In 2007 Taiga completed a property wide reconnaissance mapping Project (1:25,000 scale).

Greater detailed (1:5,000 scale) geological mapping was later undertaken to compliment the larger scale version in areas of geologic interest. During the 2008 field season, Taiga again completed a property wide mapping programme, however, at a notably smaller scale than before (1:10,000 scale).

During the 2011 field season mapping activities increased and the identification of the graphitic prospect areas led to smaller scale mapping (1:1000 scale) over areas which appeared to have the greatest graphitic potential. Areas included were Molo (Figure 3), Fondrana, Fotsy and Seta.

9.2 Trenching

No known historical trenching is documented on Molo. During the 2011 field season a number of the new graphite rich areas were trenched including Molo. Initial graphitic carbon results from the 2011 trenching were encouraging in that they showed multiple graphic horizons present in each zone, of significant widths and grades. Because of this and coupled to the size of the electro-magnetic signature, the 2012 programme focussed on Molo and an additional 22 trenches were excavated.

Additional trenching was undertaken on Molo during May of 2013 as part of a bulk sampling exercise (Figure 16). Subsequently an additional nine trenches (totalling 1,876m) have been excavated as part of the 2014 exploration programme.

Figure 16: Trenching for the Bulk Sample on Molo, May 2013

A plan map showing the positions and grades of all the trenches excavated on the Molo deposit is provided as Figure 17 below.

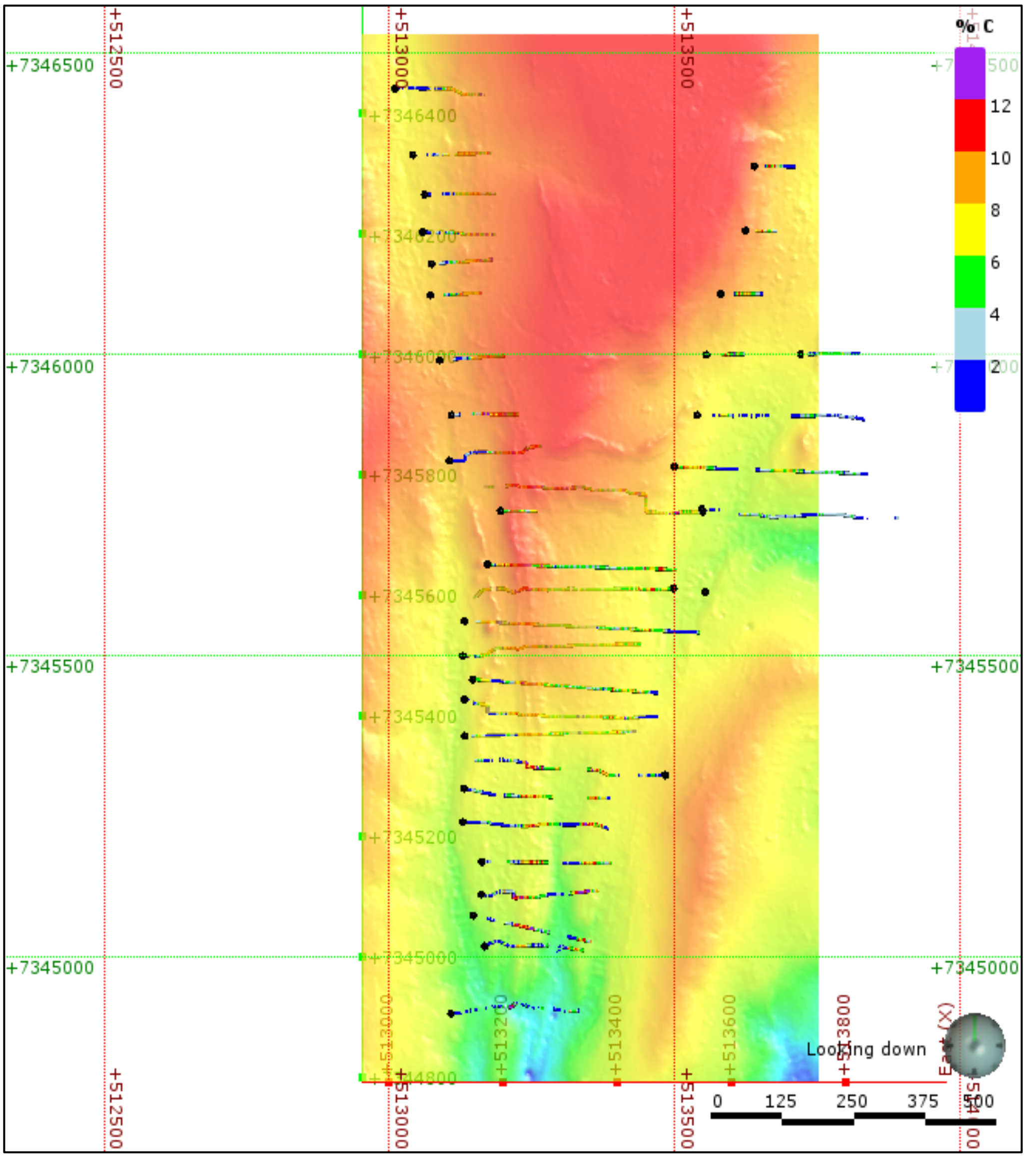


Figure 17: Plan Showing the Positions and Grades of All the Trenches Excavated on the Molo Deposit, Overlain on the Topographic Elevation Map

9.2.1 Trench Sampling

Standardized sampling methods include 2m long continuous chip samples approximately 4 cm wide being collected along the northern edge of the trench floor, consisting of about 3 kg to 4 kg of material per sample. The following procedural steps were taken during the sampling and mapping process:

Plastic sample bags were sequentially numbered with a unique series from pre-printed sample books. The Quality Assurance / Quality Control ("QA/QC") sample numbers are flagged at this point for later insertion.

The trench floor is swept clean with hand brooms to ensure there is no contamination from rubble, or fines.

Two technicians use hammers and chisels to gently dislodge the weathered rock along the channel profile.

A third technician follows behind to collect the sample material, first verifying the sample tag is in the bag, then matching the sample bag number and the sample book interval.

The sample bag was sealed with a zip tie, with the sample tag inside the bag.

Two technicians follow behind the samplers and clean / scrape the north wall of the trench to allow better visual inspection of any structures, and to remove any debris, or 'polishing' which may have occurred during excavation.

A geologist, or qualified technician using scaled paper inspects the north wall of the trench and records structures, mineralization, depth and any other notable aspects.

All samples are brought back to the camp each night for storage in the secure facility at Fotadrevo until shipment.

Samples taken from the 2012-2014 trench exploration programmes were subject to stringent QA / QC and their lengths and percentage carbon are presented in Table 11 below. This data was used in the estimation process.

Table 11: Samples taken from the 2012-2014 trench exploration programmes

Trench	From (m)	To (m)	Length (m)	C%
MOLO-TH-12-01	28	318	290	6.58
MOLO-TH-12-02	2.5	358	355.5	6.01
MOLO-TH-12-03	5	380	375	7.74
MOLO-TH-12-04	37	119	82	8.89
MOLO-TH-12-05	32	90	58	7.4
MOLO-TH-12-06	45	125	80	6.56
MOLO-TH-12-07	52	138	86	7.34
MOLO-TH-12-08	88	140	52	7.65
MOLO-TH-12-08	186	286	100	7.25
MOLO-TH-12-09	38	128	90	6.92
MOLO-TH-12-09	166	220	54	8.79
MOLO-TH-12-10	92	121	29	7.18
MOLO-TH-12-10	214	230	16	5.06
MOLO-TH-12-11	34	94	60	5.01

Trench	From (m)	To (m)	Length (m)	C%
MOLO-TH-12-11	158	194	36	5.74
MOLO-TH-12-12	84.5	257	172.5	5.68
MOLO-TH-12-13	16	52	36	4.51
MOLO-TH-12-13	74	320	246	6.55
MOLO-TH-12-14	82	176	94	7.87
MOLO-TH-12-15	84	152	68	6.94
MOLO-TH-12-16	80	96	16	4.37
MOLO-TH-12-18	38	65	27	4.88
MOLO-TH-13	0	299	299	6.14
MOLO-TH-14	44	222	178	6.32
MOLO-TH-15	27.3	112	84.7	5.88
MOLO-TH-16	32.6	108	75.4	6.82
MOLO-TH-17	0	332	332	6.15
MOLO-TH-18	20	351	331	5.58
MOLO-TH-19	24	298	274	5.37
MOLO-TH-20	88	250	162	7.13
MOLO-TH-21	59	212	153	6.93
MOLO-TH-22	48.6	119	70.4	6.57
MOLO-TH-23	3.3	69.3	66	7.26
MOLO-TH-24	27	65	38	8.09
MOLO-TH-25	26	66.6	40.6	6.51
MOLO-TH-26	18.6	46	27.4	7.63
MOLO-TH-27	24.6	50.6	26	6.86

9.3 Ground Geophysical Surveying

During 2011 an EM31-MK 2 (“EM31”) ground conductivity instrument was obtained to aid determination of the overall extents of the graphitic horizons delineated during the previous year's field mapping and prospecting activities. The EM31 geophysical tool was invaluable in delineating the extents of the graphitic zones, as well as their continuity. The Company's geotechnical team conducted a survey consisting of 100m spaced lines and 25m stations. In total a 160.5 km line of EM31 surveying was completed over five target areas, full details of which are available on the Company's web page:

(http://energizerresources.com/Projects/green-giant-graphite.html).

This data was used to plan the original (2011) boreholes drilled on Molo (Figure 19).

During the 2011 programme an EM31 geophysical survey was also conducted over the Molo deposit area concurrently with prospecting and subsequent geological mapping of the stronger graphitic zones. The survey (Figure 18) aided in outlining a zone length of over 2 km, with an aggregate EM31 measured strike length of 10 km. This, along with the confirmation of five strong graphitic horizons, supported further trenching and drilling.

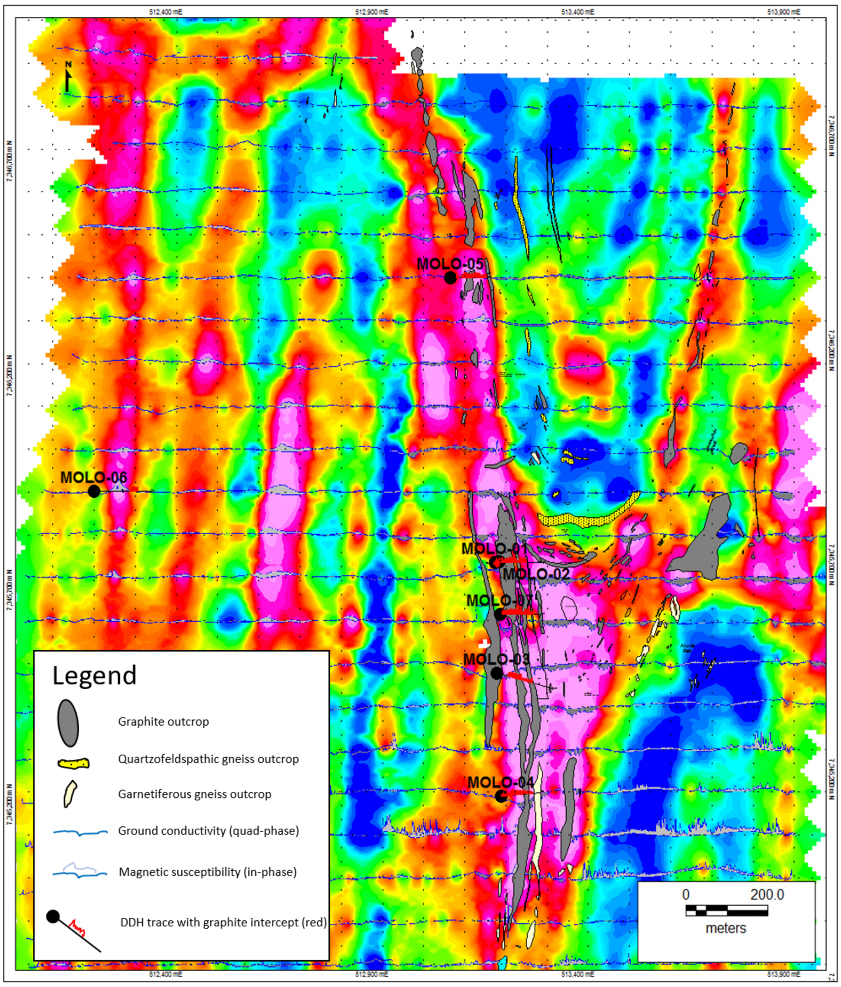


Figure 18: EM31 Geophysical Survey Map of the Molo Deposit Showing the Positions of Holes Molo-01 to Molo-07 Drilled During 2011

Bright Pink areas are interpreted graphitic areas and the grey areas represent surficial / mapped graphite outcrops. The north-eastern limb was the focus of additional work during 2014.

9.4 Prospecting and Sampling

Property scale prospecting and grab sampling was conducted during earlier exploration programmes. Prospecting typically consisted of a preliminary stage in which areas were covered on a large scale to determine Vanadium and Graphite potential. Upon discovery of any notable potential mineralisation, a larger group of prospectors were sent to the area of significance. This then allowed much more of the Property to be 'ground truthed' and, where applicable, sampled in an intensive manner to gain an understanding of all the zones.

During 2011, the Company's employees thoroughly covered the Green Giant property with special interest in graphitic showings. With the addition of the JV Property, a much larger area had to be investigated, which required helicopter assistance to access remote and marginally accessible areas. Over the course of six days the Company's technical team visited a variety of notable graphite localities including Molo. During 2012, Molo was subject to an intensive grab sampling programme, which resulted in a total of 344 samples being collected.

10 DRILLING

No known historical diamond drilling is documented for the Green Giant Property. Numerous diamond drilling programmes have been conducted over the Green Giant (and formerly the Three Horses) property in the past seven years and these have previously been covered in previous reports (Scherba and Chisholm, 2008, McCracken and Holloway, 2009; and Desautels et al., 2010). Only work directly, or indirectly pertaining to the Molo deposit is therefore covered here.

Due to the realization that a significant graphitic resource probably existed on the Green Giant-JV Property, a reconnaissance diamond drilling programme was implemented in 2011 to test the viability and potential of the graphitic prospects.

The 2011 diamond drilling commenced in early November and ran through to mid-December. During this programme the Company entered into a JV agreement with Malagasy Minerals and this phase therefore was re-focussed on the new graphitic prospects on the JV areas. This resulted in 19 holes being drilled by Boart Longyear over the Molo, Fotsy, Fondrana and Seta areas, for a combined 1701m of diamond drilled core. Of these 19 holes, one larger (PQ) diameter hole was completed and sent to Activation Labs (www.actlabs.com/) ("Actlabs") in Ancaster, Ontario, Canada for metallurgical analysis.

Of the 2011 diamond drilling, seven (Molo-01 to Molo-07) wide spaced holes were drilled on Molo. Six of these (over a strike length of 1.2 km) intersected graphitic mineralisation to a vertical depth of 75m, with down-hole thicknesses of between 60m and 150m in width. Graphite mineralisation intersected in drill core was open along strike, and at depth. Forty-one diamond drill holes, comprising 8,502.7m of diamond drilling were completed on Molo during 2012. During 2014 an additional 32 diamond drill holes (totalling 2,063m) were completed. With this most recent drill programme, a total of 80 diamond drill holes (Figure 19), (totalling 11,660m) have now been completed on Molo, and these were used for the mineral resource estimations.

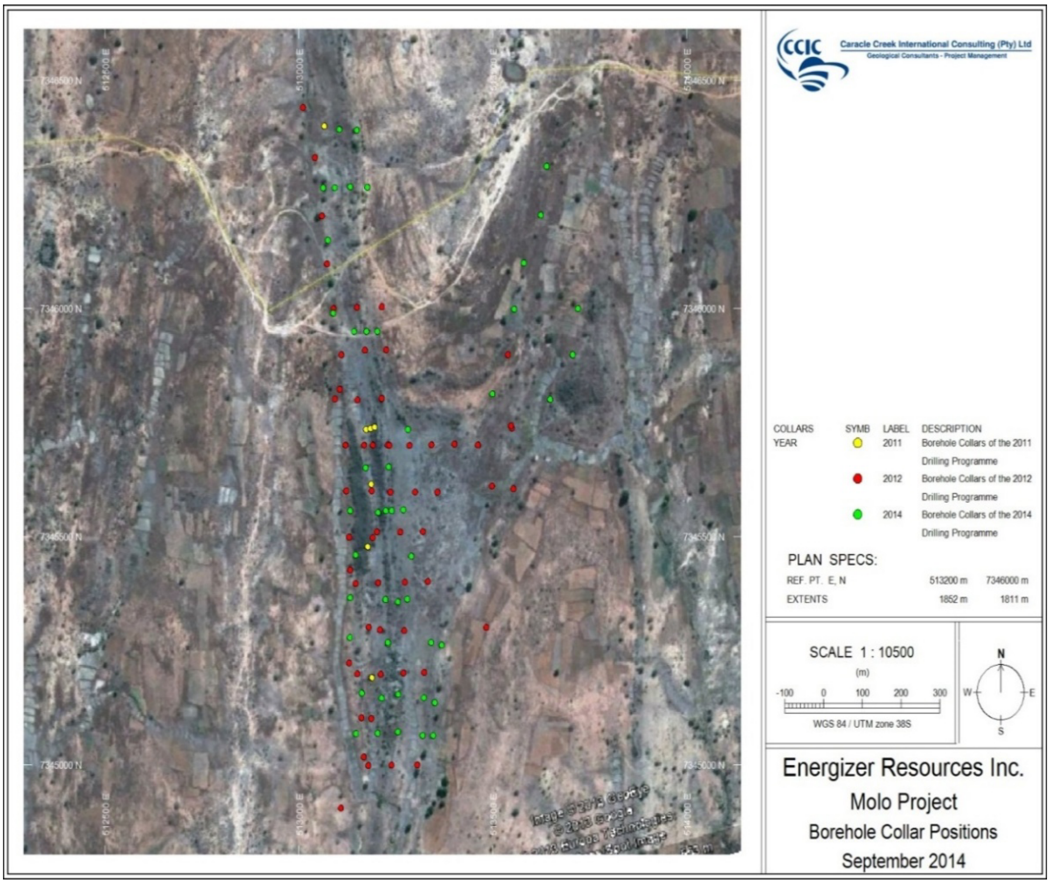


Figure 19: Borehole Collar Positions for All of the Known Boreholes Drilled on the Molo Deposit

10.1 Diamond Drill Contractor and Logistics

Between 2009 and 2014 all diamond drilling on the Green Giant-JV Project was carried out using a Boart Longyear 44 skid-mounted wire-line rig, and a Boart Longyear LF-90 skid-mounted wire-line rig, owned and operated by Boart Longyear™ (http://www.boartlongyear.com/), South Africa. The initial 70m to 100m of any borehole was generally completed with HQ core (63.5 mm diameter), and once reasonably competent rock was encountered, this was reduced to NQ (47.6 mm diameter core). On rare occasions, (as noted for the 2011 metallurgical hole) larger diameter PQ sized core (85.0 mm diameter) was extracted.

The drill moves were completed using Boart Longyear's John Deere skidder equipped with a blade and winch. Drill pads and sumps were prepared using a rental CAT 420D backhoe / loader. While drilling, all fluids were pumped directly from the sumps, with all overflow fluids directed back to the sumps.

These measures were taken to conserve drill fluids and to prevent site contamination by drill additives, or metals liberated by the drilling. Water for drilling operations was trucked from streams, creeks, or ponds sporadically located along the main drainages crossing the property and stored at the drill site until required.

10.1.1 Core Handling Procedures

Core is delivered from the Molo drill site to the Fotadrevo base camp by pickup truck at the end of every 12 hours shift, under the supervision of the drilling company, or an official designated by the Company. Drill core is stored in galvanized-steel core boxes 1m in length holding 3m PQ, 5m HQ, or 7m NQ core. The core boxes are laid out on constructed core benches in sequential order. A general review of the core is undertaken, and the core is washed, or rinsed of debris and drilling fluids. The Company's technicians then assess the overall condition and recovery of the core and complete a lithological 'quick log'. Errors in run markers are noted. All drill core is presently stored at the Company's Fotadrevo camp within a secure, 20m x 25m fenced enclosure.

10.1.2 Core Logging

The Company utilizes a logging system developed by Taiga, which has subsequently been altered slightly and has become the standard for all Company procedures. At this stage of exploration there is no restricted list of rock units for core logging as the stratigraphy is still being developed. The Company's geologists, however, attempt to utilize a standard set of units and aim to discuss and agree upon any new units prior to their utilization.

Core logging was previously recorded onto paper logs which were subsequently transferred to computer. The Company has, however, recently purchased Panasonic field laptops and thus all data is currently entered directly into these units. Core logs contain observations of geology, structure, mineralogy, alteration and sample interval descriptions.

10.1.3 Core Recovery

Trained technicians are responsible for collecting geotechnical data such as Rock Quality Description ("RQD") and core recovery. The data is recorded onto paper forms with entry into computer logs at the end of the day. The core recoveries are considered as being good and fit for resource estimation purposes.

10.1.4 Core Photography

The core is photographed in groups of two boxes (Figure 20) and then forwarded for cutting. Core is typically photographed wet.

Figure 20: Core Photographs of Borehole Molo-05 Core (Boxes 17 and 18) as Supplied to CCIC by the Company

10.1.5 Collar Survey

Borehole collar locations are initially established in the field using a hand-held Global Positioning System ("GPS") instrument. Following completion of the hole the collar locations were re-measured, also using a hand-held GPS. The nominal accuracy of these positions, as stated by the manufacturer of the GPS units, is ±3 m. The bore hole collars have not been surveyed by a registered surveyor.

All drill collar sites have been reclaimed and collars marked, with nothing left in the ground, or on the drill site. All holes are plugged and cemented to approximately one metre down the hole. Furthermore, all holes are identified by engraving the collar name into the fresh cement, which when dry is very difficult to destroy (Figure 21).

Figure 21: Concrete Marker Showing the Collar Position of Borehole Molo 12-02

10.1.6 *Down Hole Surveys*

From 2009 till present, Boart Longyear uses single shot Reflex equipment on all diamond drill holes to measure down the hole azimuths and inclinations. Measurements were taken below the level of the surface casing, (generally 10m to 15m depth), every 50m, (unless hole conditions dictated otherwise), with a final measurement taken at the end of the hole.

10.1.7 *Geotechnical Logging*

Geotechnical logging consisted of RQD measurements and core recovery calculations. The data was collected on paper forms and later transcribed into an Excel spreadsheet. RQD measurements are calculated using a minimum 10 cm core length according to the following formula:

$$RQD = Run\ Length - \frac{(\sum Pieces\ of\ core < 10\ cm)}{Run\ Length}$$

10.1.8 *Diamond Drill Core Sampling*

The methodology utilized by the Company for diamond drill core sampling was first established by Taiga during the 2008 exploration programme and then modified to accommodate specific programme requirements as needed.

The 2012-2014 diamond drill core sampling procedure may be described as follow:

Sample interval was set at a maximum of 1.5m (run length) and shortened based on lithological breaks.

Sample intervals were recorded in the drill log and in pre-printed sample books. QA/QC samples numbers were flagged at this point for later insertion.

Plastic sample bags were numbered sequentially with the appropriate sample number.

Core was cut by a technician using a clean water spray table rock saw and both halves of the sawn core was placed back in the box.

The geologist who logged the core verified the sample tag with the sample book and placed half of the cut core into the sample bag.

The sample bag was sealed with a zip tie, placed in another larger bag (i.e., double bagged) with a duplicate sample number, and a sample tag was inserted between the sample bags to mitigate against the destruction of the sample tag.

All the samples were stored in a secure facility at the Company's Fotadrevo campsite until shipment.

10.1.9 Diamond Drill Results

All results from the 2011, 2012 and 2014 diamond drill programmes are presented in Table 12 below. The authors of this Report are of the opinion that there are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the nature of the obtained samples.

Table 12: Collar Co-Ordinates, Drilled Length and Orientation for the Molo Drill Holes

BHID	EASTING	NORTHING	ELEV	LENGTH	DIP	AZIMUTH
MOLO-01	513173	7345735	553.96	166	-45	90
MOLO-02	513184	7345737	554.52	47	-45	265
MOLO-03	513177	7345478	549.88	207.5	-45	105
MOLO-04	513188	7345191	545.79	137	-45	85
MOLO-05	513065	7346400	551.60	131	-45	90
MOLO-06	512199	7345902	543.51	200	-45	90
MOLO-07	513186	7345615	550.84	142.5	-45	90
MOLO-12-01	513121	7345600	550.78	463.6	-45	90
MOLO-12-02	513185	7345601	550.58	89	-45	270
MOLO-12-03	513236	7345598	551.52	311	-45	90
MOLO-12-04	513300	7345600	553.24	221	-45	90
MOLO-12-05	513360	7345600	553.96	170	-45	90
MOLO-12-06	513150	7345400	548.59	364.5	-50	90
MOLO-12-07	513210	7345400	547.65	299	-50	90

BHID	EASTING	NORTHING	ELEV	LENGTH	DIP	AZIMUTH
MOLO-12-08	513270	7345400	547.86	212	-50	90
MOLO-12-09	513330	7345400	549.46	137	-50	90
MOLO-12-10	513151	7345199	545.65	320	-50	90
MOLO-12-11	513210	7345197	545.81	221	-50	90
MOLO-12-12	513270	7345200	544.46	182	-50	90
MOLO-12-13	513322	7345202	543.49	95	-50	90
MOLO-12-14	513177	7344998	541.12	260	-50	90
MOLO-12-15	513238	7345001	541.87	194	-50	90
MOLO-12-16	513305	7344999	543.16	80	-50	90
MOLO-12-17	513092	7345801	553.85	258.4	-50	90
MOLO-12-18	513151	7345799	554.37	132	-50	90
MOLO-12-19	513204	7345805	556.96	77	-50	90
MOLO-12-20	513089	7346000	552.99	234	-50	90
MOLO-12-21	513150	7346002	552.38	156.5	-50	90
MOLO-12-22	513215	7346007	554.77	54	-50	90
MOLO-12-23	513202	7345517	550.20	126.4	-50	270
MOLO-12-24	513245	7345513	550.21	177.5	-50	270
MOLO-12-25	513210	7345292	546.50	85.5	-50	270
MOLO-12-26	513274	7345295	545.81	186.5	-50	270
MOLO-12-27	513226	7345702	554.22	212	-50	270
MOLO-12-28	513170	7345909	554.69	84	-50	270
MOLO-12-29	513215	7345907	555.68	143	-50	270
MOLO-12-30	513188	7345713	553.67	126.8	-50	270
MOLO-12-31	513322	7345513	552.23	293	-50	270
MOLO-12-32	513119	7345697	553.26	327.5	-50	90

BHID	EASTING	NORTHING	ELEV	LENGTH	DIP	AZIMUTH
MOLO-12-33	513169	7345702	553.10	252.5	-50	90
MOLO-12-34	513232	7345702	554.33	236	-50	90
MOLO-12-35	513285	7345700	554.94	185	-50	90
MOLO-12-36	513341	7345702	554.59	200	-50	90
MOLO-12-37	513403	7345703	553.26	200	-50	90
MOLO-12-38	513465	7345706	549.95	152	-50	90
MOLO-12-39	513190	7345498	550.05	329	-70	90
MOLO-12-40	513179	7345301	546.95	326	-70	90
MOLO-12-41	513187	7345102	542.80	329	-70	90

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation, Assay and Analytical Procedures

At all times during sample collection, storage, and shipment to the laboratory facility, the samples were in the control of the Company, or their agents.

Note that no further sample preparation and analysis were conducted for this report as all required work were carried out for Molo Phase 1.

When sufficient sample material (grab, trench, or core) has been collected, the samples are trucked, or flown to the Company's storage location in Antananarivo, at all times accompanied by a Company employee. From there, samples are further shipped to either South Africa (Mintek, or Genalysis), or Canada (Activation Labs) for ICP-MS analysis.

Drill core samples collected during 2011 were directed to two major laboratories. All samples collected during Phase I of 2011 were sent to Mintek, South Africa. Samples were then tested for Carbon content (Total Organic Carbon and Overall Carbon content), as well as the full range of elements available through ICP-OES (Mintek code FA5) and XRF analysis. The elements tested included Al, Si, P, S, Cl, K, Ca, Ti, V, Cr, Mn, Fe, Co, Ni, Cu, Zn, Ga, Ge, As Se, Br, Rb, Sr, Y, Zr, Nb, Mo, Ag, Cd, in, Sn, Sb, Te, I, Cs, Ba, La, Ce, Hf, Ta, W, Hg, Tl, Pb, Bi, Th and U.

The remainder of samples collected during Phase 2 of the 2011 exploration programme were submitted for analysis to Actlabs, Canada. Samples were again submitted for analysis of Carbon content, as well as for a large range of elemental analysis.

During 2012, all samples were submitted to Intertek Genalysis (www.intertek.com/) ("Genalysis"). All work undertaken by Intertek is performed in accordance with the Intertek Minerals Standard Terms and Conditions of which can be downloaded from their web page.

All analytical results were e-mailed directly by both Genalysis and Mintek to the Green Giant Project Manager, as well as the Company's executive staff, and were posted on a secure website and downloaded by the Company's personnel using a secure username and password. Following the site inspection in May 2012, all analytical results were also e-mailed directly to Dr. Hancox (CCIC) and these were compared against the final data set as presented by the Company.

All of the laboratories that carried out the sampling and analytical work are independent of the Company.

11.2 QA/QC

In order to carry out QA/QC protocols on the assays, blanks, standards and duplicates were inserted into the sample streams. This was done once in every 30 samples, representing an insertion rate of 3.33% of the total.

11.2.1 Blanks

Since the 2009 drill programme, the Company has rigorously implemented a blank protocol. For Molo fine grained quartz sand sourced from a hardware store in Antananarivo was used as the blank material for the sampling campaign. An additional 93 blanks have been submitted during this campaign, taking the total number of blanks to 301. A detection limit of 0.05% Carbon was used for the purpose of this exercise. To verify the reliability of the blank samples, the detection limit and the blank + 2, and 3 times the detection limit was plotted against the date and Figure 22 represents the previous and current campaigns respectively). Blanks for the 2014 campaign shows that the majority of samples have concentrations that lie within the blank + 3 times detection limit threshold, with the maximum outlier being 0.57% C.

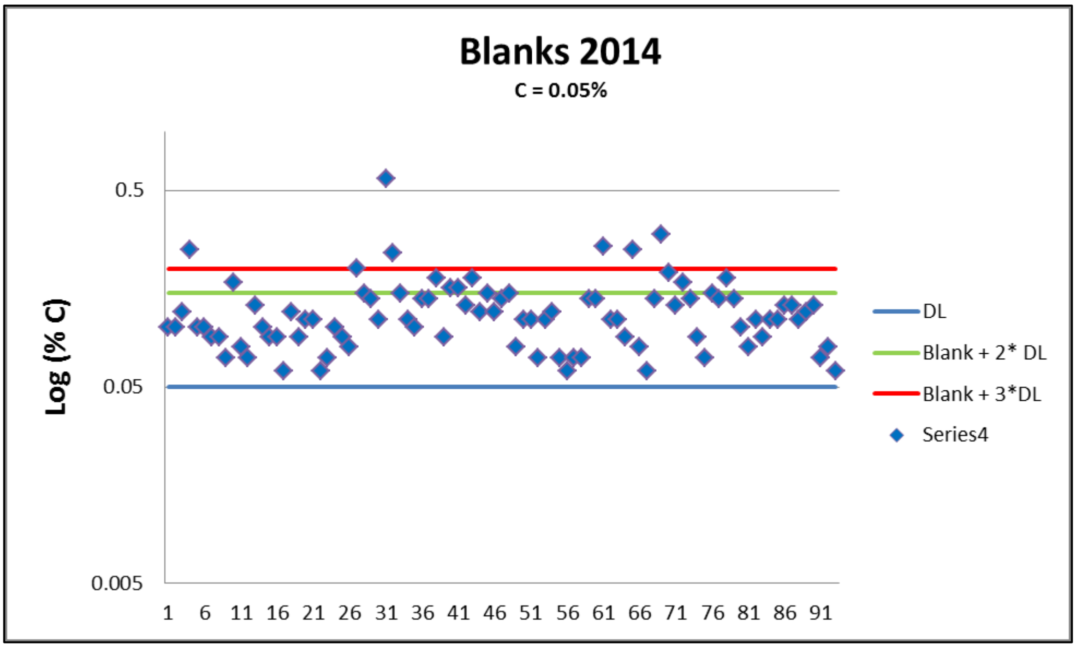


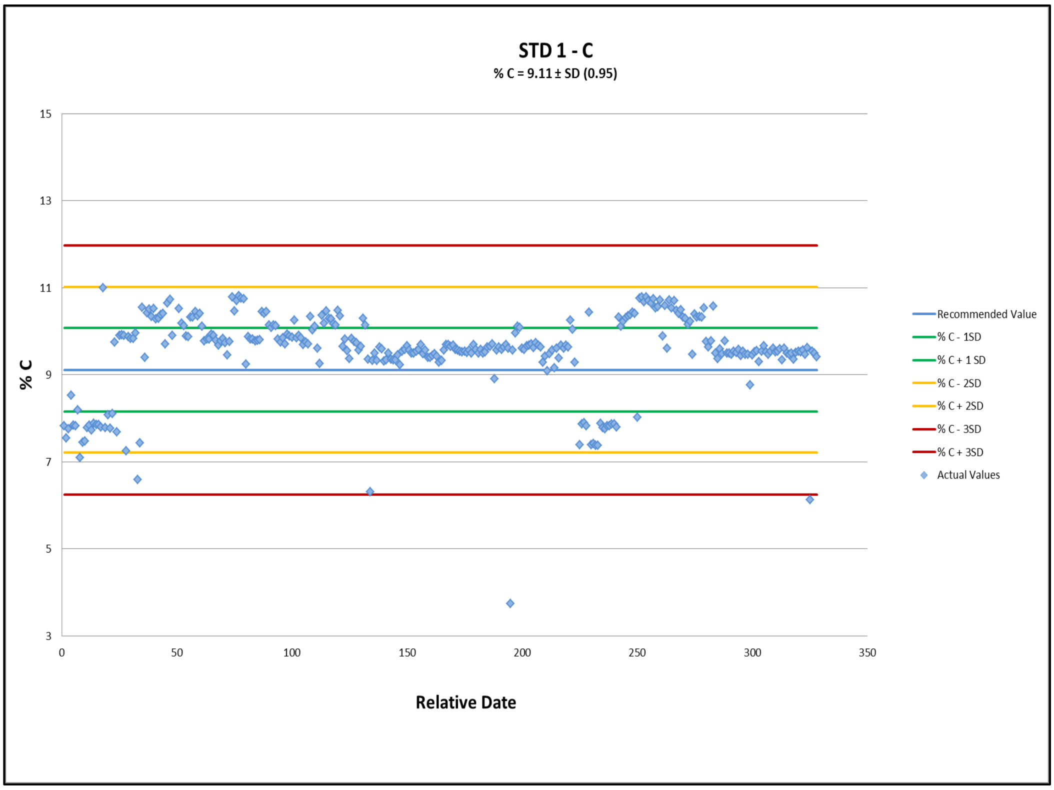
Figure 22: Plot of Log %C Versus the Date of the Analysis 2014 for the Blanks

11.2.2 Standards

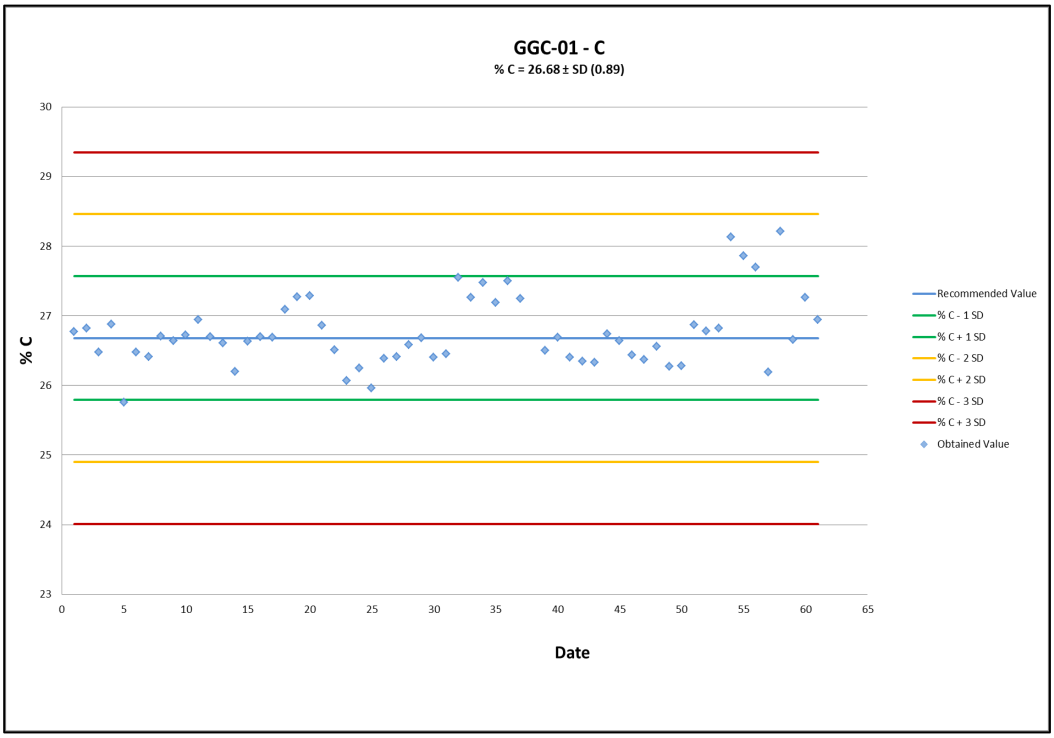
Because of the difficulty in sourcing CRM during the 2012 campaign, the Company commissioned Actlabs, Canada to create a CRM from the remaining Molo drill core pulps from the 2011 programme. As certified the Actlabs standard (STD 1 C) has a recommended value of 9.11% Carbon. For the 2014 campaign, the Company sourced two additional CRM's from GEOSTATS (Proprietary) Limited ("GEOSTATS"), namely GGC-01 and GGC-07. The recommended values for GGC-01 and GGC-07 are 24.97% C and 0.56% C respectively.

To check the reliability of the standard, a plot of the recommended CRM value versus date was created (Graph 1, Graph 2 and Graph 3).

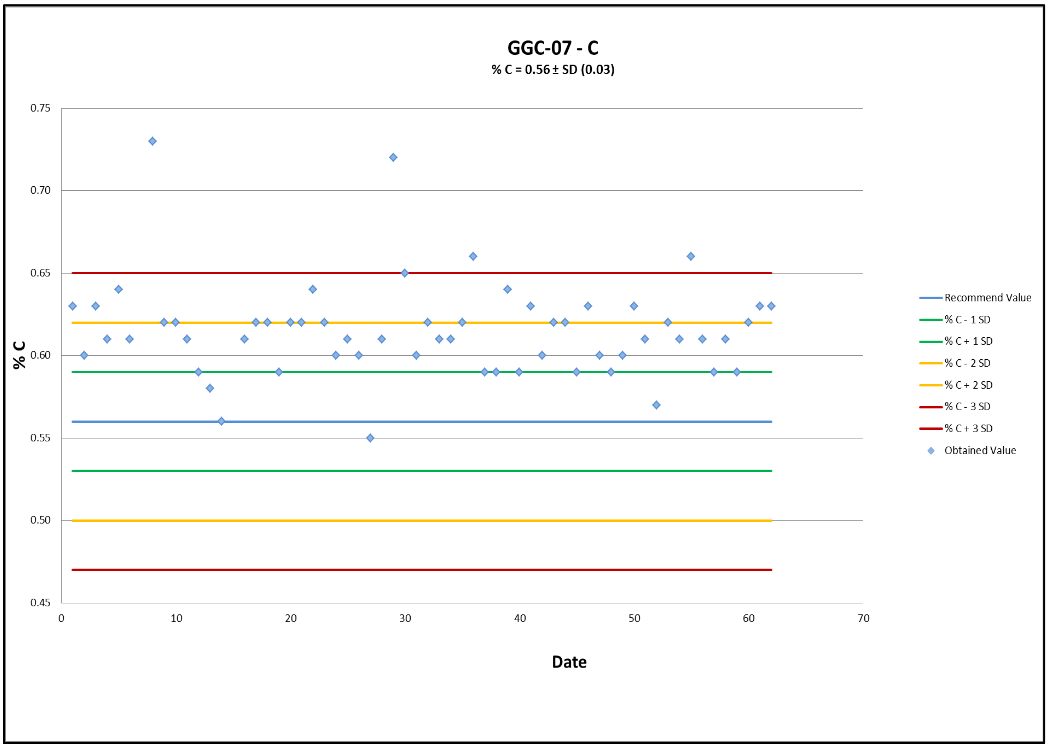
The upper and lower limits of one, two and three times the standard deviations of the recommended value are also included in the plot. For STD 1 C, all but two results fall outside the acceptable limit of three times the standard deviation. It is, however, worth noting that the obtained results indicate a slight positive bias when compared to the recommended value. For GGC-01, all the obtained results occur within two standard deviations. For GGC-07, four samples occur outside of three standard deviations. This CRM has a mean value of 0.56% C whilst the mean of the obtained values is 0.61% C, indicating a positive bias at low concentrations.



Graph 1: Graph Showing Carbon Concentration as Analysed in Std 1C



Graph 2: Graph Showing Carbon Concentration as Analysed in GGC-01



Graph 3: Graph Showing Carbon Concentration as Analysed In GGC-07

11.3 Field Duplicates

A total of 254 duplicate samples were submitted. To check how close these were to the original samples, a plot of the original samples with a zero, five, and ten per cent difference of the original samples was created. (Figure 23) 2012 and 2014 samples are shaded in blue and red respectively. The majority of the samples are within the 10% difference limit. The plot also shows a good correlation between the original value and the duplicate, as is evident from the regression line with an R2 value of 0.96.

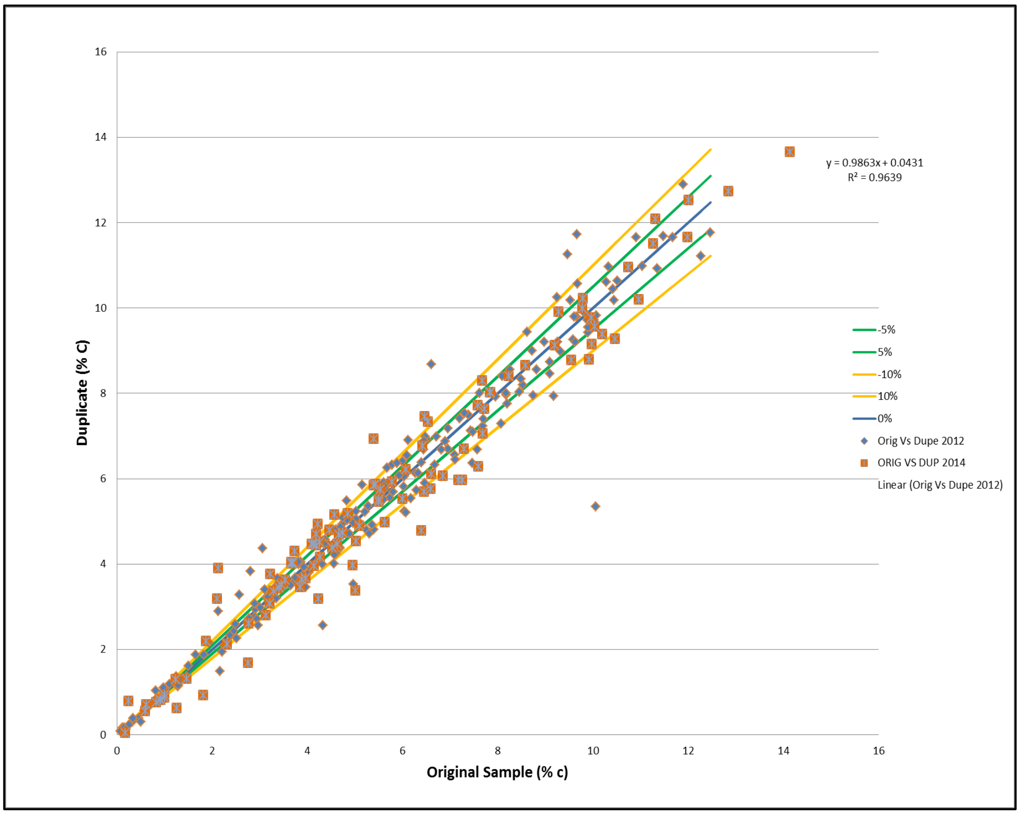


Figure 23: Original (Orig) Versus Duplicate (Dupe) Plots

11.4 Relative Density

No additional samples were collected for density measurement during the 2014 drill programme. A total of 226 relative density measurements were contained in the database presented to CCIC for Molo.

The process to measure the relative density is as follow:

Pieces of whole core were collected, and the rock types documented.

The selected section of core is then dipped into wet paraffin wax and allowed to dry. This seals the core to avoid the absorption of moisture.

The pieces of core are weighed dry, followed by weighing in a water bath.

All data is collected on paper forms and transferred to a spreadsheet for future calculations.

12 DATA VERIFICATION

Prior to CCIC's involvement with the Company and the Project, all information published regarding the 2011 exploration programme was reviewed by an independent Qualified Person as it became available. The database received by CCIC from the Company contained 80 drill holes totalling 11,660m and data from 35 trenches totalling 8,492m. With regards to the database, CCIC performed various tests to verify the integrity of the collar co-ordinates, logging and sampling procedures, and assay results. Leapfrog™ Geo software (www.leapfrog3d.com/) ("Leapfrog") was used for most of the checks.

12.1 Collar and Down Hole Surveys

During a site visit in 2014, Desmond Subramani randomly selected four drill hole collars to validate. All four drill hole collars were physically located and plotted within the accuracy of the handheld GPS unit being used for validation. While on site, the Company was in the process of undertaking a topographic survey and a DGPS re-survey of all drill hole collars.

To verify the correct position of the re-surveyed drill hole collars with respect to their elevation, collar co-ordinates were plotted against the surface re surveyed topography of the area (Figure 24). The results showed that the re-surveyed collars were within a 25 cm of the surface topography, and therefore all collar co-ordinates were deemed to be correct and were used for the geological modelling.

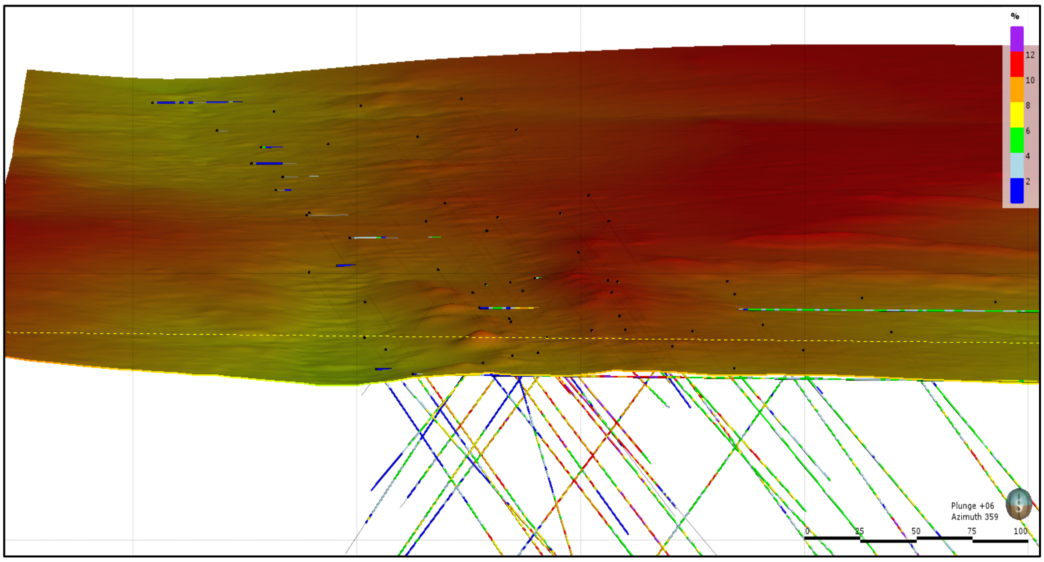


Figure 24: East-West Cross-Section Showing the Collar Positions with Respect to Topography

12.2 Drill Logs

During the initial 2012 site visit Dr. Hancox randomly selected two of the 2011 drill holes (Molo-07 and Fotsy-06) to review the log's data against the drill core (Figure 25). The holes were check logged to verify that the intervals in the logs matched the drill core. No discrepancies were observed. Molo-04 and Molo-05 were also examined.

Figure 25: Borehole Fotsy-06 - One of the Bore Holes Check Logged during May 2012 Site Visit

For the 2014 drilling campaign, a CCIC Geologist (Mr W. Ngangolo) supervised the drilling, logging and sampling. Drill hole logs were checked in the field, prior to uploading into the Company's Database, managed by Eric Steffler. Dr. Schneiderhan also undertook various check logs during here 2014 site visit.

Database checks undertaken in the Leapfrog™ software ("Leapfrog") included, (but were not limited to) gaps, consistency in the logging codes, and overlaps in the depth 'From' and 'To' entries. No gaps were encountered in the database. There seemed to be some lack of consistency in the sage of the logging codes as initially the graphite bearing unit was termed as either gneiss (Gn) coding, or graphitic gneiss, (coded as GfGn). This was later edited by the Company and the graphite bearing gneiss was finally coded GpGn.

Whilst a few irregularities were encountered with the data supplied, the authors are of the opinion that following on the validation and verification checks, the data is adequate for the level of geological modelling and resource estimation undertaken.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A series of metallurgical and mineralogical investigations was completed at Mintek in South Africa and SGS Lakefield in Canada on samples originating from the Molo deposit amongst others, which included Fotsy and Fondrana deposit material in the case of the Mintek test work. The primary purpose of the programs from the perspective of the Molo material was to:

Develop a robust process flowsheet that produces a combined graphite concentrate grading of at least 95% total carbon.

Demonstrate the robustness of the proposed flowsheet for the expected mill feed during the first few years of operation.

Generate concentrate and tailings samples for downstream evaluation by potential vendors and off-take partners.

The first scoping level program was completed at Mintek in 2012. This was followed up with a flowsheet development program at SGS Lakefield in June to August 2013. The proposed process flowsheet was employed in a pilot plant campaign, again conducted at SGS Lakefield, in September / October 2013 to confirm the robustness of the flowsheet and to generate concentrate for downstream testing.

A process optimization program was conducted at SGS Lakefield between June and September 2014 to simplify parts of the circuit. This optimized process flowsheet was validated in a concluding variability flotation program between October 2014 and January 2015. All process flowsheet optimization was conducted under the assumption that the average ore mined per annum would be 856,071 T at a head grade of 7.04% C. The resulting flow sheet was subsequently tested for higher head grades and is therefore deemed suitable for both Phase 1 and Phase 2 development.

The key economic factors for a graphite Project from a metallurgical point of view are graphite recovery, flake size distribution and concentrate grade.

While there is market for graphite concentrates grading as low as 80% carbon, the price of a product increases with carbon grade. For a flotation graphite concentrate without further purification a product grading between 94% and 97% total carbon is typically targeted.

Large graphite flakes demand higher prices due to a limited supply on the market, while concentrates containing graphite flakes smaller than 200 mesh (-75 microns) are available in abundance and, therefore, create a much lower revenue on a per tonne of concentrate basis. Consequently, a process development program must focus on flake size preservation to maximize the amount of medium and large flakes in a concentrate, while minimizing the percentage of small flakes.

It is pertinent that the decisions made in process development programs consider these economic factors. While not all of them can be optimized at the same time, a balanced approach is required.

13.2 Flowsheet Development – Mintek

A scoping level metallurgical and mineralogical program was completed on five composites at Mintek in South Africa in 2012. These composites originated from the Fondrana, Fotsy, and Molo zones. Only the results for the Molo composite are summarized in this section, which was specified as trench sample Molo-TH-11-01.

The Molo composite graded 10.6% carbon. Three rougher kinetics tests were completed on the composite with a primary grind size of P50 of 150 microns. A carbon recovery of almost 99% was achieved on average with rougher tailings grades of 0.1% to 0.2% carbon. The rougher concentrate grades yielded 53.5% to 59.0% carbon. The reagents used in the program were illuminating paraffin as the graphite collector and Dowfroth 200 as the frother.

Two cleaner tests using conditions that were previously established for Fotsy and Fondrana composites produced concentrate grades of 77.4% carbon with the dispersant sodium silicate and 77.5% carbon without sodium silicate at carbon recoveries of 95.9% to 97.5%.

A final cleaner flotation test was conducted using the flowsheet depicted in Figure 26. The second regrind was conducted in a pebble mill with smaller media with a size range of 4 mm to 10 mm instead of the regular pebble mill containing +20 mm gravel. The concentrate grade improved to 82.7% carbon at 92.9% carbon recovery because of this secondary regrinding and cleaning circuit.

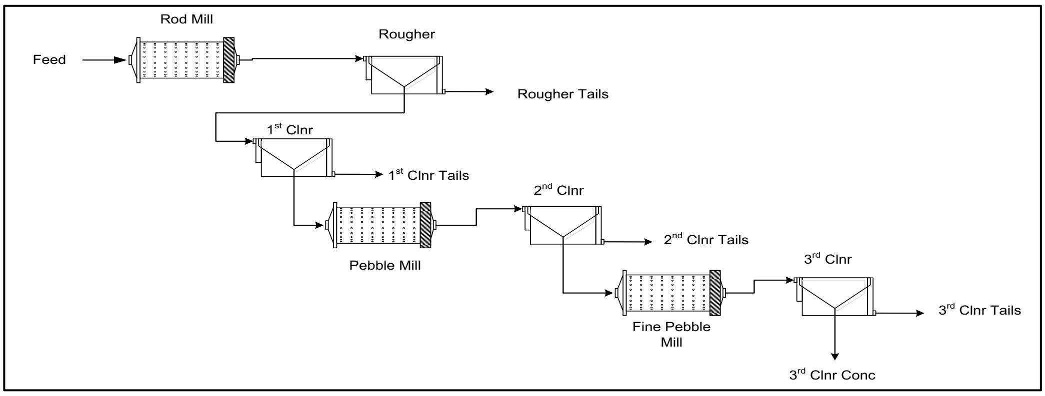


Figure 26: Flowsheet for Molo Mineralization – Mintek

It should be noted that the Mintek test work program focused on carbon recovery optimization and did not focus on flake size preservation.

13.3 Flowsheet Development – SGS

A full flowsheet development program was subsequently initiated at the SGS Lakefield site in June 2013 using a high-grade and a low grade composite from the Molo deposit. These composites were generated by collecting a sub-sample from the 200t bulk trench sample that was generated for pilot plant testing. For every 2m of the total trench length of 160m, approximately 2.5t of material were extracted for piloting. A 5 kg sub-sample was then removed from each 2.5t sample to form the high grade and low grade composite for laboratory scale testing.

A series of eight rougher and cleaner flotation kinetics tests evaluated the flotation performance of each of the two composites, as well as a 50:50 blend of the two composites.

The flotation approach employed a flash flotation stage on test charges that were stage crushed to minus 6 mesh followed by primary grinding and rougher flotation. The cleaner tests employed a primary polishing grind followed by three stages of cleaner flotation.

Since the blended composite produced good metallurgical results, a decision was made to use this composite for all further development work as this maximizes the mineral resource of the deposit and simplifies mining.

The following eight rougher kinetics tests evaluated primary grinding with conventional steel rods and ceramic media.

The energy input created by the ceramic media proved insufficient in a primary grinding application and, therefore, steel media was chosen with grind times targeting a mill discharge of P_{80} = 400 to 500 microns.

Four proceeding cleaner tests investigated the conditions of the primary cleaning circuit.

The flash and rougher concentrates were combined and subjected to a polishing grind using ceramic rods.

The polishing times were varied between 7 minutes and 30 minutes.

The polishing mill discharge was upgraded in three stages of cleaner flotation and the third cleaner concentrate was subjected to a size fraction analysis.

A primary polishing time of 22 minutes was identified as the best compromise between maximizing the intermediate concentrate grade and minimizing flake degradation.

Considering the liberation properties of the different flake sizes, the intermediate concentrate was classified at 80 mesh (177 microns) and 150 mesh (106 microns) and each classification product was then subjected to a secondary polishing grind and cleaner flotation.

The remaining four tests F22 to F25 in the flowsheet development program investigated the impact of varying secondary polishing times on the combined concentrate grade. Secondary polishing times of 6 minutes, 8 minutes, and 45 minutes for the +80 mesh, +150 mesh, and -150 mesh size fractions, respectively, achieved the best metallurgical results.

The proposed flowsheet including classification sizes and polishing grind times used in test F24 is depicted in Figure 27 and the size fraction analysis results of the combined concentrate are presented in Table 13. The conditions of test F24 were selected as they constituted the best compromise between maximizing concentrate grades while minimizing graphite flake degradation. All size fractions yielded concentrate grades of at least 95.2% total carbon including the smallest size fraction of -400 mesh (-37 microns).

The mass recovery into the large flake category of +80 mesh (177 microns) was very good at 47.6% and only 18.6% of the mass reported to the fines smaller than 200 mesh (74 microns).

The carbon recovery into the combined concentrate was 83.4% in this open circuit test. Graphite recovery is expected to increase during closed circuit operation as the intermediate streams are circulated rather than treated as tailings.

The significant differences between the metallurgical results obtained in the Mintek and SGS programs were the result of several factors. Firstly, the two Molo composites were collected from different areas of the deposit. Secondly, the Mintek approach focused on maximizing carbon recovery into the graphite concentrate. In contrast, the SGS approach focused on maximizing flake size preservation and concentrate grade.

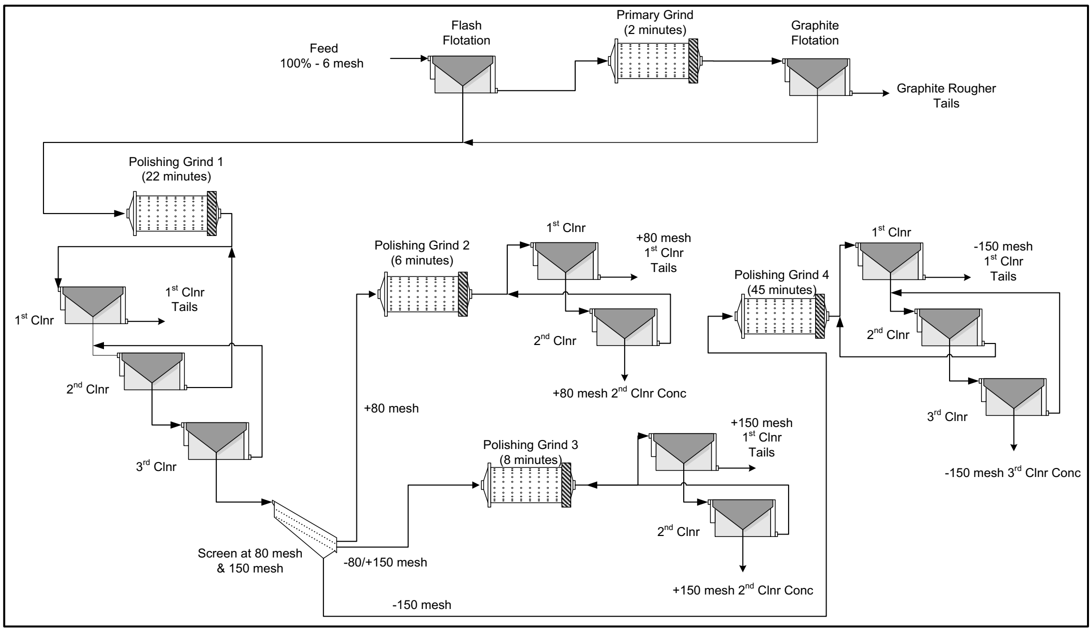


Figure 27: Proposed Molo Mineralization Flowsheet – SGS

Table 13: Size Fractions Analysis Results of Test F24

Product		Mass (%)	Grade (% Total Carbon)
Mesh	Microns		
+48	+297	21.7	97.4
-48/+65	-297/+210	17.9	96.7
-65/+80	-210/+177	8.1	95.7
-80/+100	-177/+149	10.9	96.0
-100/+150	-149/+106	12.5	95.2
-150/+200	-106/+74	10.3	95.3
-200/+325	-74/+44	8.5	96.2
-325/+400	-44/+37	3.0	95.9
-400	-37	7.1	95.7

13.4 Pilot Plant Campaign

A pilot plant campaign using a 200t Molo bulk (trench) sample was conducted at the SGS Lakefield, Canada site in September / October 2013.

The pilot plant campaign was carried out to confirm the robustness of the above SGS proposed flowsheet that was developed in the laboratory program. Further, approximately 10.8t of graphite concentrate were generated for downstream testing including vendor testing and evaluation of potential off-takers.

The bulk sample that was processed in the pilot plant was collected on site by extracting two samples of approximately 100t each, one of the low grade area and one of the high grade area of the Molo deposit. The aim for the two bulk samples was to be representative of the future plant feed.

The position of the bulk sampling trench is depicted in Figure 28. The first 80m starting from the west were classified as high-grade material and the following 80m as low grade material. The trench sections were sub-divided into 2 m intervals and approximately 2,500 kg of ore were extracted from each interval after removal of any soil and overburden. At the same time, approximately 5 kg were removed from each interval to generate the material for the SGS laboratory scale flowsheet development program described above

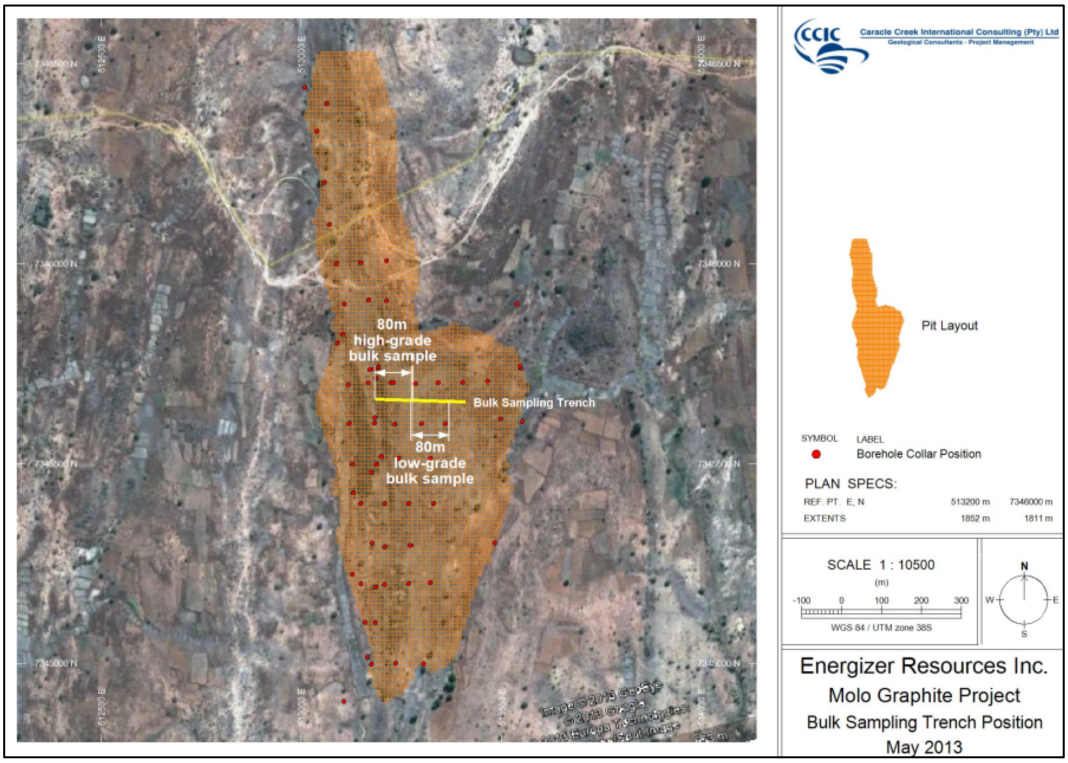


Figure 28: Map Showing Position of Bulk Sampling Trench

A single composite was processed in the pilot plant campaign grading 7.98% total carbon. The pilot plant composite was generated by blending high grade and low-grade ore in a ratio of approximately 46:54.

The results of comminution tests that were carried out on the individual composites, as well as the pilot plant blend are summarized in Table 14.

Table 14: Summary of Comminution Test Results for Pilot Plant Composites

Sample	Relative	JK Parameters			RWI	BWI	AI
Name	Density	A x b[1]	A x b[2]	T_a	(kWh/t)	(kWh/t)	(g)
Blend	2.29	199	0	2.28	8.3	11.2	0.129
High Grade	2.08	-	151	1.88	8.7	11.4	0.125
Low Grade	2.37	-	192	2.09	6.9	9.7	0.106

[1] A x b from DWT

[2] A x b from SMC

A total of nineteen pilot plant runs, PP-01 to PP-19, were completed to process the approximately 200t of the Molo bulk composite using the flowsheet depicted in Figure 29. The only reagents that were employed were fuel oil number 2 (diesel collector) and methyl isobutyl carbinol (MIBC frother) at average dosages of 117 g/t and 195 g/t, respectively.

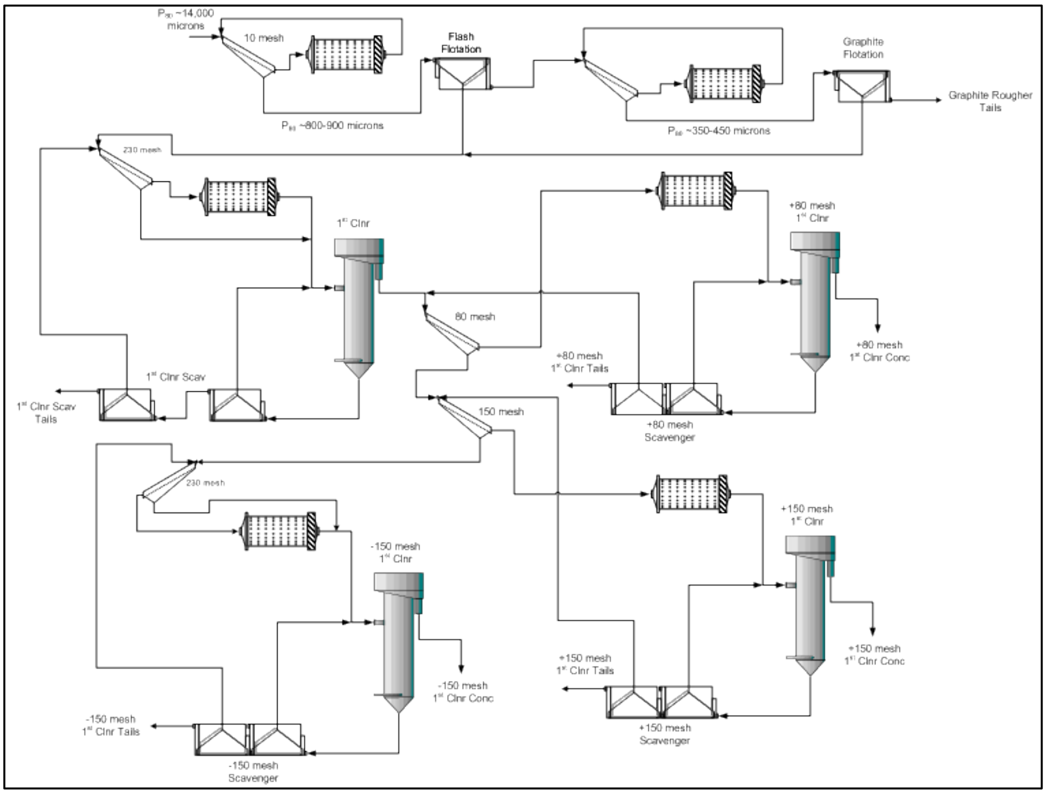


Figure 29: Molo Pilot Plant Flowsheet

A total of fifteen (15) circuit surveys were completed between PP-04 and PP-18. In each survey, up to 36 streams were sampled 5 times over the course of an hour and then submitted for sizing and/or chemical analysis.

All products submitted for chemical analysis were assayed for total carbon and the low grade tailings streams also for graphitic carbon.

The results were then used to generate full circuit mass balances using the BILMATTM data reconciliation software.

In addition, hourly grab samples of strategic streams were collected and submitted for chemical analysis, or sizing.

Turn-around times of sizing and assay results were typically less than one hour. This approach was chosen to ensure that metallurgical targets were met and to facilitate the optimization of the operating conditions.

The average feed sizes to the flash and rougher flotation circuits were P_{80} = 835 microns and P_{80} = 443 microns, respectively.

The results from pilot plant test PP-17B were selected to generate the process design criteria. The combined concentrate grade yielded 93.1% total carbon at a carbon recovery of 90.6%.

A summary of the mass balance is presented in Table 15.

Table 15: Summary of Mass Balance for Pilot Plant Test PP-17B

Product	Mass (%)	Assay % Total Carbon	Distribution % Total Carbon
+80 mesh Concentrate	4.0	95.9	53.1
+150 mesh Concentrate	1.7	92.7	21.9
-150 mesh Concentrate	1.3	85.1	15.7
Combined Concentrate	7.0	93.1	90.6
Combined Tailings	93.0	0.72	9.4
Plant Feed	100	7.17	100

The product size of the final graphite concentrate from the fifteen surveys yielded an average value of P_{80} = 268 microns.

The average size distribution and total carbon grade of each size fraction are presented in Table 16.

Table 16: Average Combined Concentrate from Fifteen Pilot Plant Surveys

Size Mesh	Size Microns	Mass as Percentage of Total Concentrate Mass in %	Grade % Total Carbon
+48	+297	15.7	97.7
+65	+210	17.6	97.4
+80	+177	10.2	96.7

Size Mesh	Size Microns	Mass as Percentage of Total Concentrate Mass in %	Grade % Total Carbon
+100	+149	9.7	96.4
+150	+105	15.0	96.1
+200	+74	10.1	95.2
-200	-74	21.6	88.2

The metallurgical results from the laboratory scale flowsheet development program on the 50:50 high-grade / low-grade composite as well as the results from the laboratory and pilot scale testing of the actual pilot plant composite are summarized in Table 17.

While there were some differences in the results, the data from the three test phases correlated well overall.

The largest difference was the carbon grade of the -150 mesh size fraction in the pilot plant, which was approximately 5% lower compared to the two lab results. It was postulated that this was likely the result of poor polishing efficiency due to dewatering difficulties with the -150 mesh cleaning circuit feed.

Table 17: Comparison of Laboratory and Pilot Plant Metallurgical Results

Product	Laboratory Scale		Pilot Plant
	Master Composite (F24)	Pilot Plant Composite	Pilot Plant Composite
% Mass > 80 mesh	47.6	42.4	43.5
% C(t) > 80 mesh	96.8	96.8	97.4
% Mass -80/+150 mesh	23.5	25.3	24.7
% C(t) -80/+150 mesh	95.6	95.6	96.3
% Mass -150 mesh	28.9	32.3	31.8
% C(t) – 150 mesh	95.7	95.7	90.4

13.5 Optimization and Metallurgical Variability Program

The flowsheet that was employed in the pilot plant campaign provided a fair degree of flexibility to operate the circuit on a relatively ad-hoc basis to meet specific, prevailing market demands in terms of product quality. However, with a focus on reducing capital and operating costs, as well as improving the ease of operability of the plant, an optimization program was initiated to evaluate the possibility of simplifying the process flowsheet while maintaining the graphite concentrate quality. In May 2014, a review of the metallurgical work completed by Mintek and SGS was accordingly conducted by representatives of DRA and the Company, in conjunction with the author of this chapter.

The first process option identified consisted of the original front end of the original flowsheet with flash and rougher flotation stages. The combined flash and rougher concentrates are subjected to a polishing grind followed by a cleaner flotation circuit.

The intermediate cleaner concentrate is classified on a screen and the screen oversize constitutes a final graphite concentrate. This necessitates that the primary polishing and cleaner flotation conditions can produce a flotation concentrate grading at least 95% total carbon in the larger size fractions.

The screen undersize comprising of below target concentrate is subjected to a secondary cleaning circuit with a polishing mill and cleaner flotation. The cleaner concentrate from this circuit and the screen oversize then constitutes the final combined graphite concentrate.

The second process option was further simplified by eliminating the graphite rougher flotation circuit by incorporating flash flotation only. The flash flotation concentrate is then subjected to a polishing grind and cleaner flotation.

The graphite concentrate generated in this primary cleaning circuit constitutes the final concentrate. This highly simplified process option is based on the postulation that the degree of liberation in the flash concentrate is superior to a rougher concentrate as it contains mostly large graphite flakes that generally are more easily liberated and upgraded in the primary cleaning circuit. By eliminating the rougher circuit, the middlings of graphite and gangue minerals in the primary cleaning circuit feed are reduced significantly, which require more mechanical manipulation to improve mineral liberation.

The increased graphite losses to the flash tailings associated with the simplified flowsheet may be offset by reduced capital and operating costs and a superior graphite concentrate in terms of flake size distribution.

13.5.1 Sample Selection

Based on the sample locations and logs provided by CCIC, DRA in conjunction with geologists from CCIC selected quarter core samples from various locations and depths within the Molo deposit for optimization and variability test work. Twenty (20) drill core samples were selected based on pit location, depth and indicated grade. The samples that were selected are summarized in Table 18.

Table 18: Sample for Optimization and Variability Testing

Description	Drill Core Identification
Year 1-5 High Grade Material	Molo 13, 14, 15, 16, 17, 18, 34, 35 and 36
Year 1-5 Low Grade Material	Molo 45, 46, 47 and 48
Year 5+ North Pit High Grade Material	Molo 37, 38 and 39
Year 5+ South Pit High Grade Material	Molo 29, 30, 31 and 32

As part of the optimization program six comminution composites were generated using drill core from different depths within the Molo 2015 FS 5-year pit layout. Four (4) comminution composites were tested at SGS and two (2) additional composites were shipped to Mintek in South Africa for testing. The make-up of the six (6) comminution composites is shown in Table 19.

Table 19: Comminution Composites – SGS and Mintek

Lab	Description	Depth (m)	Drill Core Identification
SGS	Comminution Composite #1	14-28	Molo 16, 17, and 18
SGS	Comminution Composite #2	57 - 85	Molo 46
SGS	Comminution Composite #3	14 -28	Molo 34 and 35
SGS	Comminution Composite #4	0-14	Molo 29, 30 and 32
Mintek	Grindmill Shallow Composite	0-14	Molo 15, 18, 35, 36, and 39
Mintek	Grindmill Deep Composite	28-56	Molo 15, 29, 37, and 45

13.5.2 Comminution Testing

A summary of the results of the comminution tests conducted at SGS is presented in Table 20 and reveals that the ore is typically very soft or soft with low abrasivity. Only the Bond ball mill grindability tests at the smaller screen size of 212 microns produced indices that placed the ore into the medium hard category.

Table 20: Summary of Comminution Tests – SGS

Sample Name & Depth of the Drill Hole Intervals	Relative Density	JK Parameters		RWI (kWh/t)	BWI (kWh/t) 500 µm	BWI (kWh/t) 212 µm	AI (g)
		Axb	Ta				
Comminution Composite #1 – Medium	2.25	147.6	1.70	9.3	12.1	14.4	0.097
Comminution Composite #2 – Deep	2.36	157.8	1.73	7.1	8.8	12.9	1.116
Comminution Composite #3 – Medium	2.23	126.0	1.46	9.3	12.1	13.2	0.081
Comminution Composite #4 – Shallow	2.29	299.9	3.38	6.0	8.0	12.1	0.030

Two composites containing drill core from the shallow and the deep areas of the Molo 2015 FS 5 year pit layout were shipped to Mintek in South Africa for further Bond ball and rod mill tests, as well as Grindmill tests. The results of these comminution tests are summarized in Table 21. The Bond rod and ball mill test results are consistent with the results obtained for the four composites tested at SGS Lakefield.

Table 21: Summary of Comminution Test - Mintek 2014

Sample Name	Grindmill Feed Top Size -6.7mm		Grindmill Feed Top Size -9.5mm		RWI (kWh/t)	BWI (kWh/t) 500 µm	BWI (kWh/t) 212 µm
	% Passing 425 µm at 3 kWh/t	% Passing 150 µm at 10 kWh/t	%Passing 425 µm at 3 kWh/t	% Passing 150 µm at 10 kWh/t			
MOLO 0-14	89.93	69.81	83.95	70.41	7.2	9.3	13.0
MOLO 28-56	77.43	65.38	76.68	68.05	9.2	10.9	13.7

13.5.3 Optimization Flotation Program

A series of six optimization composites were generated to evaluate the two alternative flowsheet options. The primary two composites were 50:50 blends of high grade and low grade mineralization from the shallow section (0m to 14m depth) and the deep section (28m to 56m depth) of the deposit.

Two drill holes of each the high-grade (HG) and the low-grade (LG) mineralization were selected to generate these two composites. A rougher kinetics test was completed on the two composites and the results revealed that the composite from the shallow section produced an inferior metallurgical response.

Since the process plant must be able to treat all ore within the pit layout, a decision was made to proceed with the majority of the optimization program using the more challenging shallow composite.

A series of rougher kinetics tests on the 50:50 LG:HG shallow and deep (F3 to F12) composites was completed to establish the primary grind size required to achieve a combined flash and rougher carbon recovery of 94% to 95%. This grind size was established at P_{80} = 400 to 450 microns.

A series of four open circuit cleaner flotation tests evaluated the possibility of obtaining a flotation concentrate grading at least 94% total carbon with a single polishing and cleaning circuit.

The four tests, (F13 to F16), included a flash flotation circuit only followed by polishing and cleaning to determine if the improved liberation properties of the flash concentrate would result in reduced polishing and cleaning requirements. Even at the longest polishing time tested, the combined concentrate graded only 85.1% carbon.

Consequently, the simplest proposed flowsheet consisting of flash flotation, polishing, and a single cleaning circuit proved insufficient to produce target concentrate grades even if lower graphite recoveries were accepted by employing flash flotation only.

Three open circuit cleaner tests, (F17 to F19), with flash and rougher flotation, polishing grind and cleaner flotation evaluated the polishing time necessary to achieve satisfactory concentrate grades in the coarser size fractions, which tend to display improved liberation properties. A polishing time of 30 minutes proved sufficient to generate an intermediate flotation concentrate that yielded grades more than 95% total carbon in the size fractions larger than 48 mesh (297 microns).

The intermediate cleaner concentrate was classified on a 48-mesh screen and the screen oversize constituted a final concentrate. The screen undersize was subjected to different polishing times followed by secondary cleaning in two cleaner flotation tests (F20 and F21). Both tests failed to produce a final concentrate grade of at least 95% total carbon.

In order to evaluate whether these metallurgical properties were created by an individual sub-sample of the optimization composite, each of the four drill hole intervals included in the optimization composite was subjected to a batch cleaner test using the conditions of test F21, which employed a secondary polishing time of 60 minutes.

The two low grade composites from drill holes Molo-45 and Molo-47 produced high combined concentrate grades of 97.4% total carbon and 97.7% total carbon, respectively.

In contrast, the two high grade composites from drill holes Molo-16 and Molo-35 yielded lower combined concentrate grades of 83.6% total carbon and 89.0% total carbon, respectively.

To evaluate the variation of metallurgical results for the different composites, a decision was made to proceed with variability flotation testing to develop a better understanding of the metallurgical processing properties.

13.6 Variability Flotation – Phase I

A variability flotation program was carried out in October 2014, using twenty different drill hole composites, which are specified in Table 22.

Table 22: Variability Composites – Phase I

Drill Hole	Depth of Drill Hole Interval		
	0-14m	14-28m	28-56m
MOLO-14	√	√	√
MOLO-16			√
MOLO-17	√	√	√
MOLO-30	√	√	√
MOLO-35			√
MOLO-38	√	√	√
MOLO-46	√	√	√
MOLO-48	√	√	√

The optimized flowsheet and conditions shown in Figure 30 were chosen for the variability flotation program. Each variability composite was subjected to the flowsheet using this flowsheet with minor adjustments to the primary grind time, flotation times and reagent dosages.

These adjustments were required to address the different hardness of the various composite, as well as observations made during the test with regards to the flotation response.

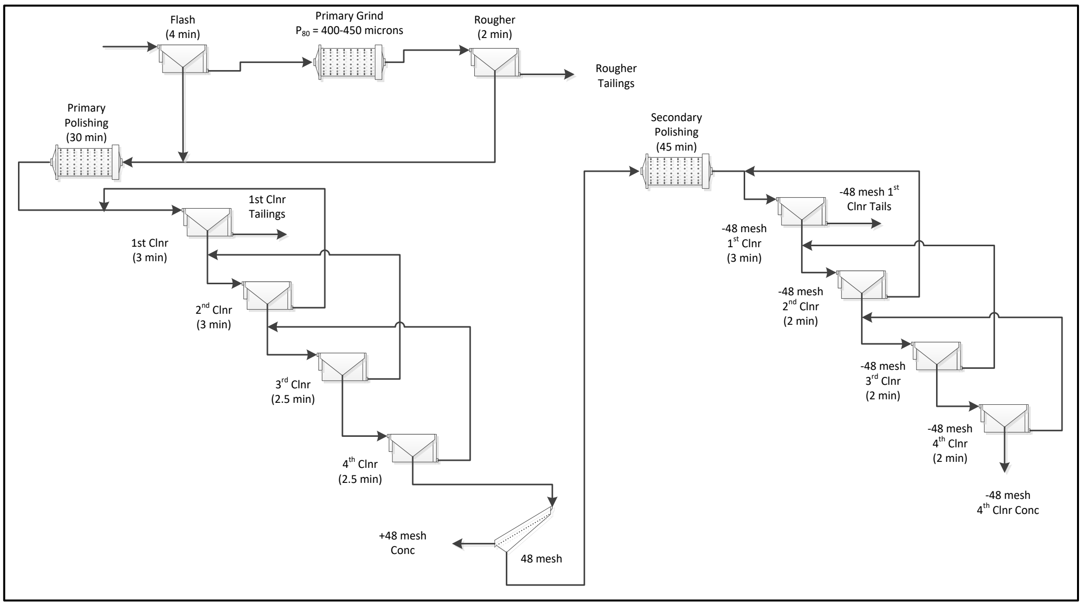


Figure 30: Optimized Flowsheet

Each of the twenty variability composites was subjected to an open circuit flotation test. The mass recovery into the various size fractions is summarized in Figure 31, alongside the average pilot plant results. The chart reveals that the average flake size distribution of the twenty variability composites compared well with the average results of the pilot plant.

The mass recovery into the +48 mesh and +65 mesh size fractions was up to 3.2% better for the variability samples. The range of mass recovery into specific size fractions was significant and the largest for the +48-mesh product.

The lowest mass recovery into the jumbo flake category was 7.4% for the Molo-46 (0m to 14m) composite while the highest mass recovery into this product of 34.9% was achieved with the Molo-48 (0m to 14m) composite.

An analysis of different potential factors such as grade and depth of the composite did not produce a strong relationship between the variable and the flake size distribution.

The average mass recovery into the +80 mesh size fractions of concentrates from high-grade composites was 47.4% compared to 46.1% for the low-grade composites.

With regards to depth, the average mass recovery into the +80 mesh size fractions of concentrates from shallow (0m to 14m) and medium (14m to 28m) depth composites was almost identical at 44.6% and 44.4%, respectively. Only the concentrate from deep (28m to 56m) composites contained a slightly higher percentage of +80 mesh material at 50.2% mass recovery.

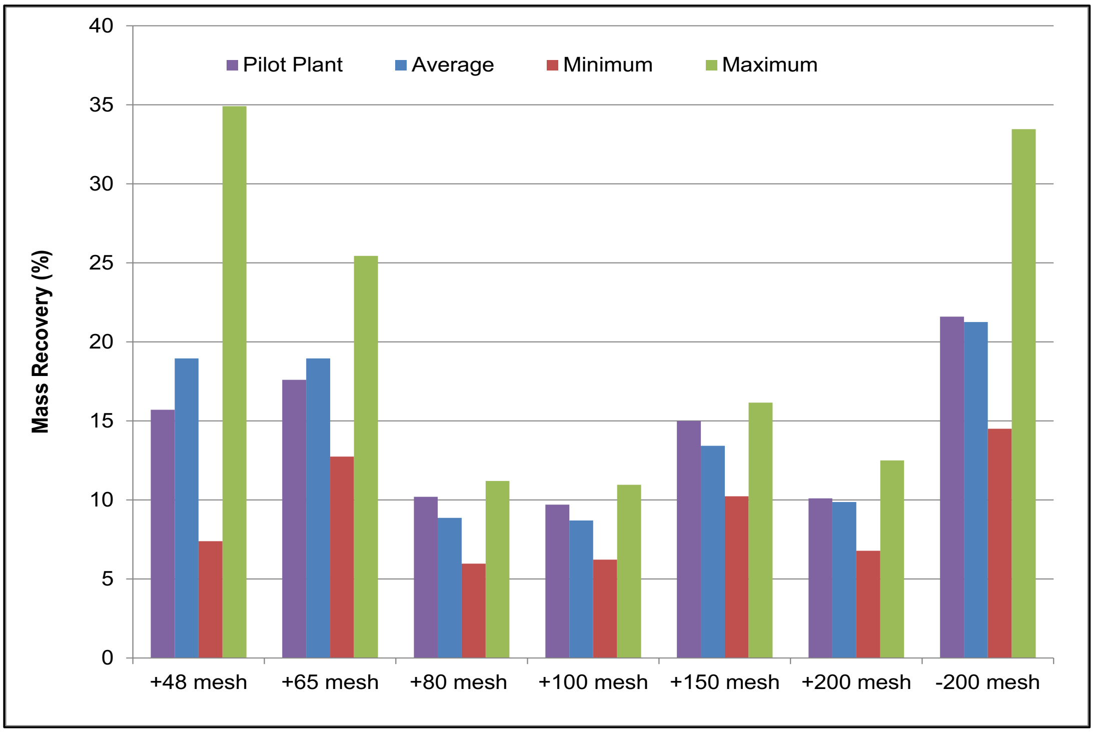


Figure 31: Range of Mass Recovery into Size Fractions (V1 to V32 and Pilot Plant)

The average, minimum, and maximum total carbon grades of tests V1 to V32 for each size fraction are depicted in Figure 32 together with the average pilot plant results. While the maximum grades matched, or exceeded those of the pilot plant, the average concentrate grade was up to 3.5% total carbon lower compared to the pilot plant for all size fractions greater than 200 mesh.

The average grade of these size fractions was less than 94% total carbon and as low as 93.1% total carbon in the -65/+80 mesh size fraction. The minimum grades were 90% total carbon, or less for all size fractions greater than 200 mesh.

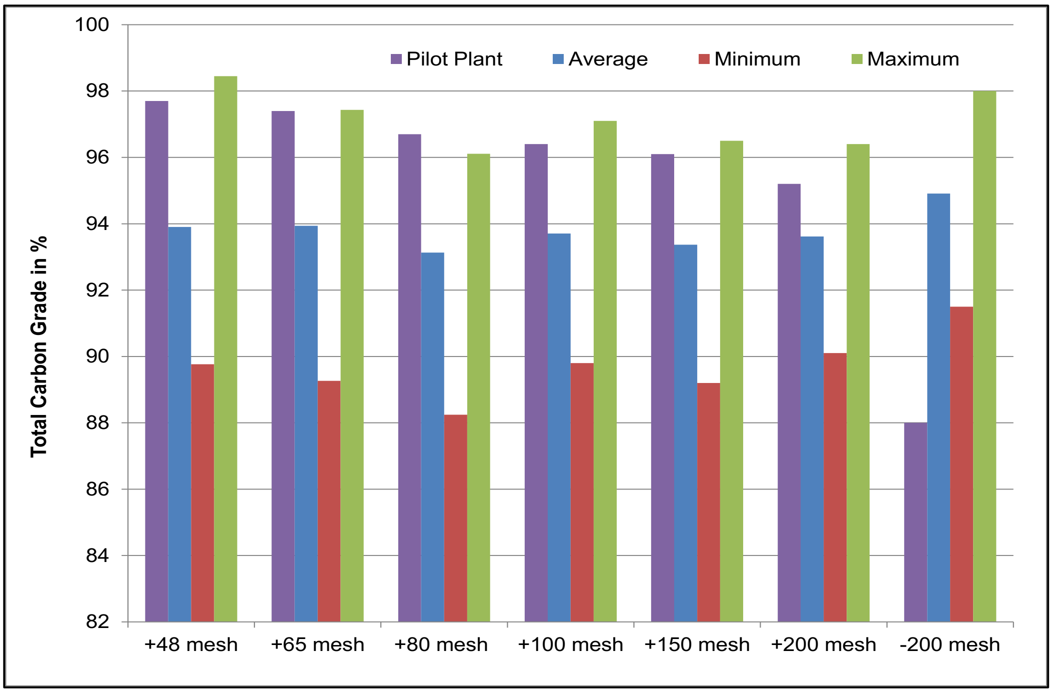


Figure 32: Range of Carbon Grades of Size Fractions (V1 To V32 and Pilot Plant)

The variability in the concentrate grade of the variability composites is further illustrated in Figure 33 which depicts the combined concentrate grade of each of the thirty-two (32) variability flotation tests.

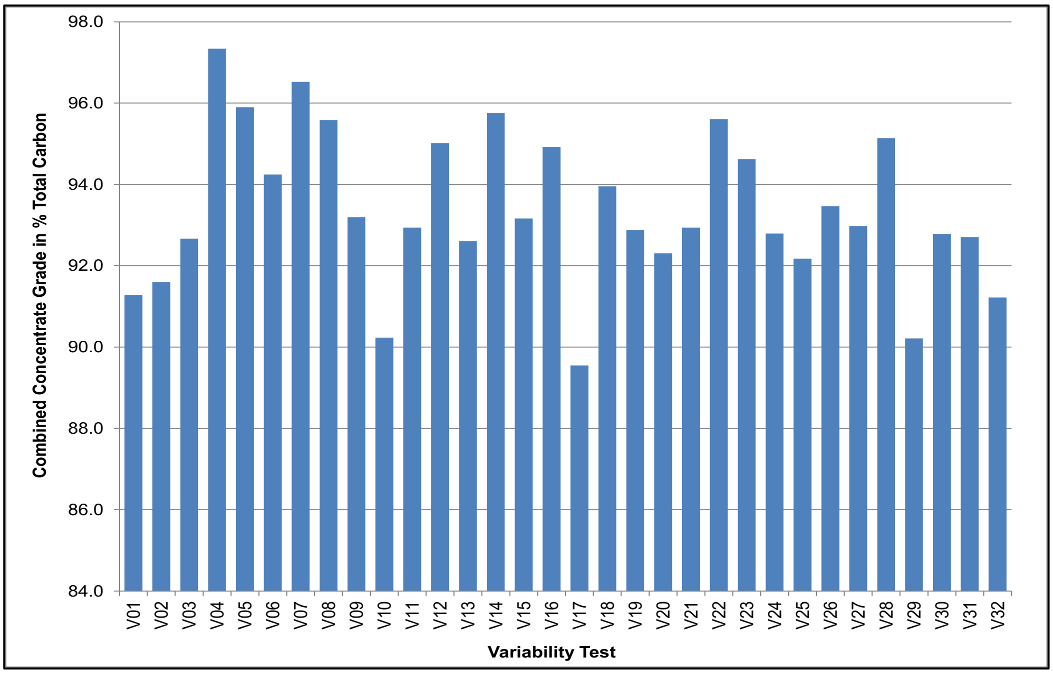


Figure 33: Combined Concentrate Grade of Tests on Variability Composites (V1 To V32)

The location and the metallurgical response of the drill hole intersections that were evaluated in the variability program within the Molo 2015 FS 5 year pit layout of the original

53,000 tpa Molo 2015 FS are depicted in Figure 34. The pit outline is demarked by the blue line. The three intersections for each drill hole represent the depth intervals 0m to 14m, 14m to 28m, and 28m to 56m starting from top to bottom.

The colour coding of each depth interval was conducted based on the legend in the figure. The results show that only seven drill hole intersections within the Molo 2015 FS the 5 year pit layout achieved a concentrate grade of 94% total carbon, while five composites produced grades of greater than 92% and less than 94% total carbon and two composites graded less than 92% total carbon.

The remaining six composites that were tested fell outside the 5 year pit layout, but the metallurgical response was consistent with the other composites.

The analysis reveals that origin of the composites within the depth interval of the drill hole does not appear to have an impact on the metallurgical response as the three grade ranges were identified in each the three depth intervals.

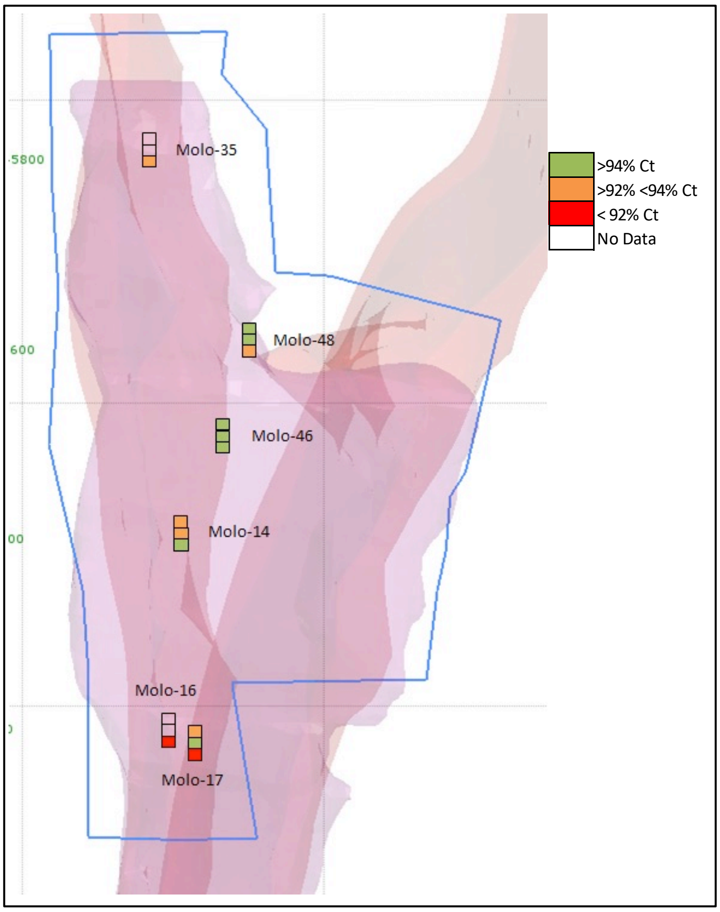


Figure 34: Location and Performance of Phase I Variability Drill Holes

The results obtained in this variability program revealed that the metallurgical response was inconsistent and appears independent of the location of the drill hole and sampling depth.

One limitation of the variability program was the fact that the six drill holes were located on a north-south axis through the deposit and did not cover the east-west extent. In discussions with the Project geologist, it was postulated that most of these drill holes originated from a transition zone between high grade and low grade ore and that this transition zone may be responsible for the inferior flotation properties.

It was paramount to determine if the large variability was linked to this potential transition zone, or if it is encountered throughout the entire mineral resource. If the inferior flotation response is only encountered in a limited and relatively small area, material from this area can be blended with other ore to achieve the flotation concentrate grade target. However, if the substantial variability is encountered throughout the entire mineral resource, upgrading strategies for the concentrate will have to be explored, which would likely result in the addition of an upgrading circuit at the tail end of the proposed process flowsheet. Hence, a decision was made to proceed with a second phase of variability flotation testing.

13.6.1 Optical Mineralogy

In order to develop a better understanding of the lower concentrate grade achieved on some of the variability composites, samples of concentrates grading only 90% to 92% total carbon were submitted for basic optical mineralogy examination.

The optical mineralogy revealed that graphite occurred less than 50 microns to 1,000 microns in size. The graphite was generally free but contained impurities of non-sulphide gangue (NSG) minerals.

The NSG minerals were fine-grained between 10 microns and 200 microns, but were inter-layered with graphite across its entire length.

One important observation was that less than 1% of the NSG minerals occurred as liberated grains. An example of inter-layered graphite particles is displayed in Figure 35. The non-sulphide minerals indicated by the green arrows were inter-layered with graphite (red arrows).

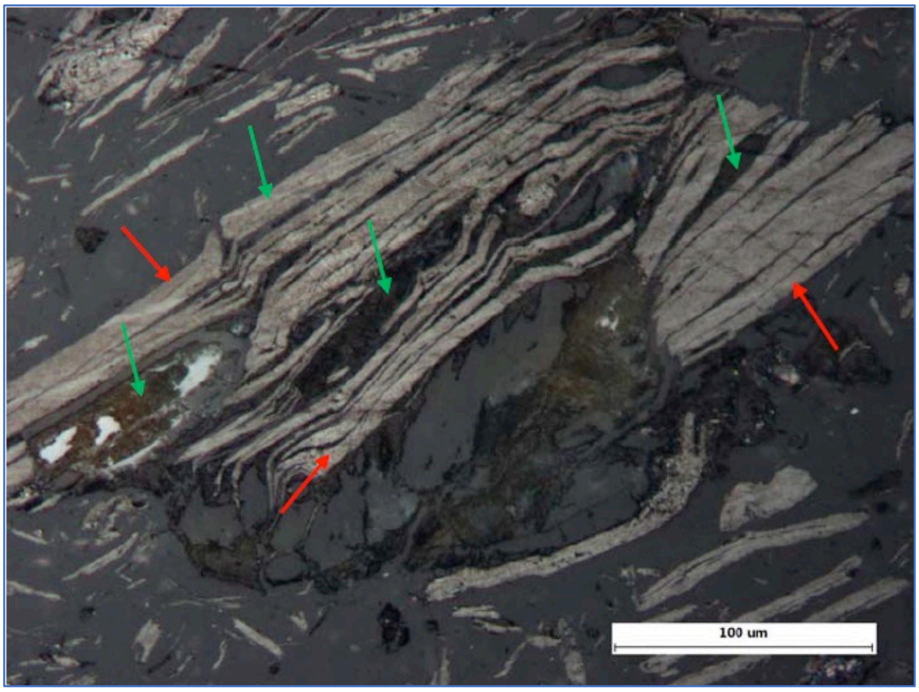


Figure 35: Intercalated Graphite

13.7 Variability Flotation – Phase II

A second variability flotation program was completed on the Molo deposit on a number of drill holes and area composites due to inferior metallurgical results obtained during an optimization and initial variability program. The primary reason for the inferior concentrate grade that was encountered in the flotation program was inter-layering of graphite and non-sulphide gangue minerals. It was postulated that this may be linked to a transition zone between high-grade and low-grade ore that was identified by the Project geologists. A second phase of variability testing was conducted in the program documented in this report using drill core that originated within the 5 year mine pit boundaries.

A total of 3.5 tonnes of core was received at the SGS Lakefield site in December of 2014 and included all drill holes from the 2012 and 2014 campaign that fell within the 5 year mine pit perimeter. The drill core was prepared to generate 21 drill hole composites and 15 area composites. The average grade of all composites was 7.04% total carbon and 6.54% graphitic carbon.

13.7.1 Drill Hole Composites

A total of seven drill holes with three depth intervals were subjected to open circuit cleaner tests to evaluate the metallurgical response of the Molo mineralization in the eastern and western areas of the Molo 2015 FS 5-year pit layout.

The drill holes used in this phase of testing were generated in a drilling campaign that was conducted in 2012. Although the core was stored for more than two years and exposed to potential oxidation. It was concluded that any degradation of the core would likely not have an impact on the metallurgical response of the graphite. This assumption was made based on the results from the first phase of variability flotation testing, which did not identify a statistically significant difference in the metallurgical response of weathered shallow material and fresh deep core.

A list of composites that were subjected to the variability flotation tests is shown in Table 23.

Table 23: Variability Composites – Phase II

Drill Hole	Depth of Drill Hole Interval		
	0-14m	14-28m	28-56m
MOLO 12-05	√	√	√
MOLO 12-09	√	√	√
MOLO 12-21	√	√	√
MOLO 12-26	√	√	√
MOLO 12-33	√	√	√
MOLO 12-37	√	√	√
MOLO 12-38	√	√	√

The same flowsheet and conditions as in the Phase I variability program were employed in the twenty-one open circuit flotation tests.

The location and metallurgical performance of the Phase 2 drill hole composites are depicted Figure 36 and confirms Phase I results.

Good and poor performing composites were frequently encountered in the same drill hole in adjacent depth intervals. Based on the metallurgical results of the individual drill hole composites, no specific area could be characterized as consistently good, average, or inferior performing.

The results suggest that further treatment of the concentrate is required. Repeat tests with longer polishing times using the existing flowsheet failed to produce improved concentrate grades in most cases.

Optical mineralogy that was conducted on concentrates from the Phase 2 variability program confirmed the metallurgical challenge of intercalated graphite, which will require a different more aggressive upgrading approach than the standard polishing mill grinding.

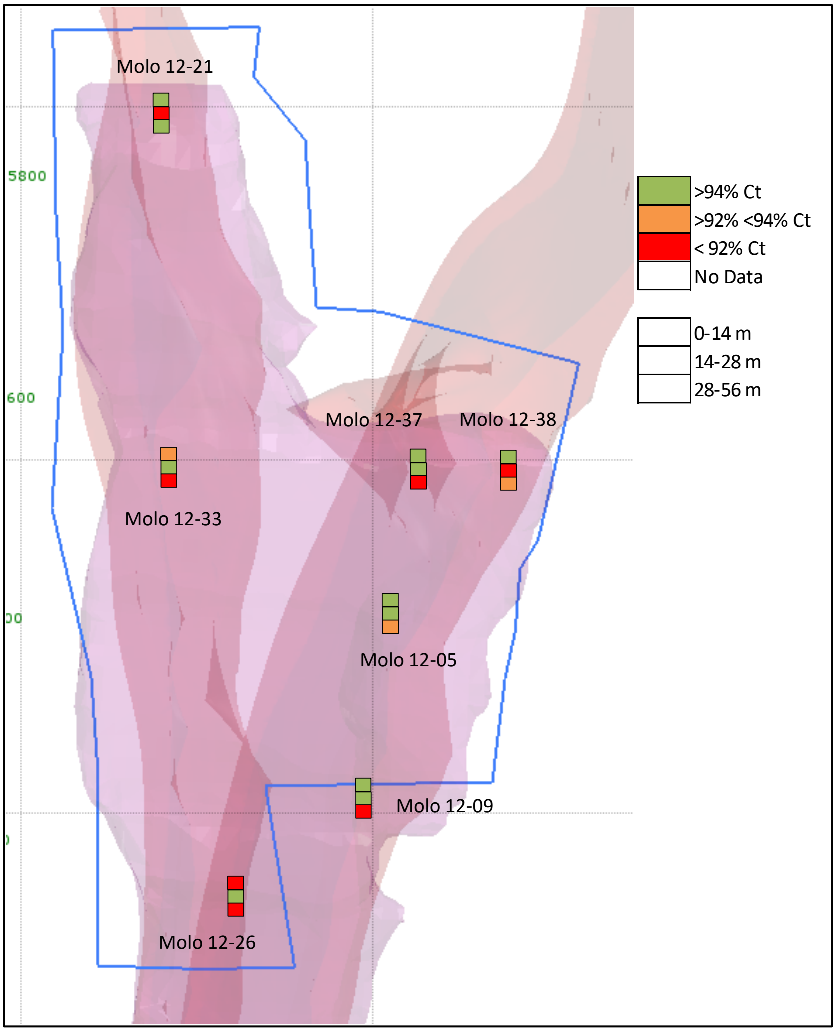


Figure 36: Location and Performance of Phase I Variability Drill Holes

13.7.2 Area Composites

In an attempt to develop a better understanding of the average metallurgical response of the Molo mineralization, area composites were generated, which included samples from all drill holes that were available.

For this purpose, the Molo 2015 FS five (5) year mine pit layout was split into five areas and each area composite was then generated by combining sub-samples from all drill holes that fell into a specific area.

The three depth intervals 0m to 14m, 14m to 28m and 28m to 56m were maintained for the area composites ie. a total of fifteen area composites were generated. A summary of the drill holes that were included in each area composite is provided in Table 24.

Table 24: Drill Holes included in Area Composites

Composite ID	Drill Hole ID's
Area Composite 1	MOLO 12-20, 12-21, 12-28, 12-29
Area Composite 2	MOLO 12-01, 12-02, 12-03, 12-18, 12-19, 12-27, 12-30, 12-32, 12-33, 12-34, MOLO-01
Area Composite 3	MOLO 12-04, 12-05, 12-35, 12-37, 12-38
Area Composite 4	MOLO 12-23, 12-24, 12-31, 14-15
Area Composite 5	MOLO 12-07,12-08, 12-09, 12-26, 12-40, 14-17, MOLO-22

The average flake size distribution of the fifteen area composites is shown in Table 25. For comparison purposes, the average flake size distribution of the pilot plant campaign and the two variability programs are presented in the same table.

The data reveals good agreement between the results, which attests to the robustness of the flake size distribution across the Molo mineralization.

Table 25: Comparison of Flake Size Distribution

Screen Size	Area Composites	Variability Phase I	Variability Phase II	Pilot Plant Campaign
+48 mesh	26.5	19.0	26.6	15.7
+65 mesh	17.0	19.0	18.3	17.6
+80 mesh	8.1	8.9	8.3	10.2
+ 100 mesh	6.6	8.7	6.6	9.7
+150 mesh	12.2	13.4	12.0	15.0
+ 200 mesh	8.4	9.9	8.2	10.1
- 200 mesh	21.2	21.3	19.9	21.6

The grades of the combined concentrates of the fifteen area composites are presented in Figure 37. Only two of the composites produced concentrate grades of greater than 94% total carbon.

Six composites graded between 92% and 94% total carbon and the remaining seven composites produced concentrates of less than 92% total carbon.

The best performing composite was that of Area 3 (0m to 14m) with a concentrate grade of 95.0% total carbon, while the worst performing composite was Area 4 (14m to 28m) which produced a concentrate grading of only 89.4% total carbon.

The average concentrate of all fifteen composites was 92.1% total carbon.

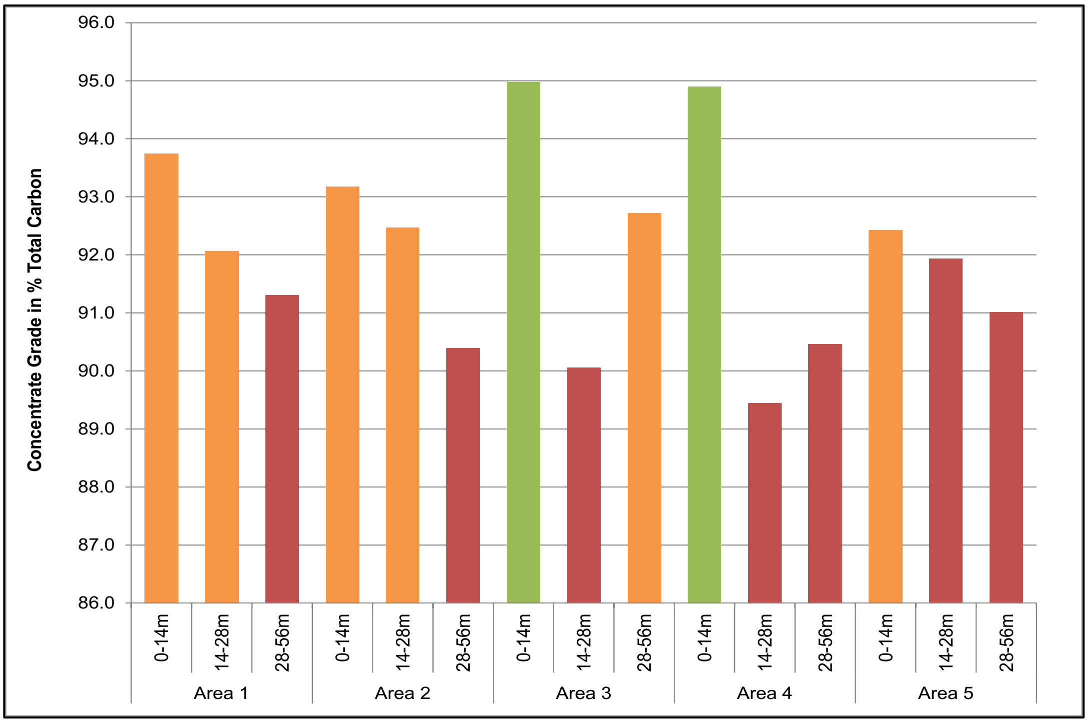


Figure 37: Combined Concentrate Grades of Twenty-One Area Composites

The location of the five areas and the metallurgical response of the fifteen area composites are shown in Figure 38. Only the top 14m of the Area 3 and Area 4 composites produced good concentrate grades, which is consistent with the results of the individual drill holes that originated from that area and depth interval.

The postulation that further upgrading of the graphite flotation concentrate is required was confirmed for the area composites.

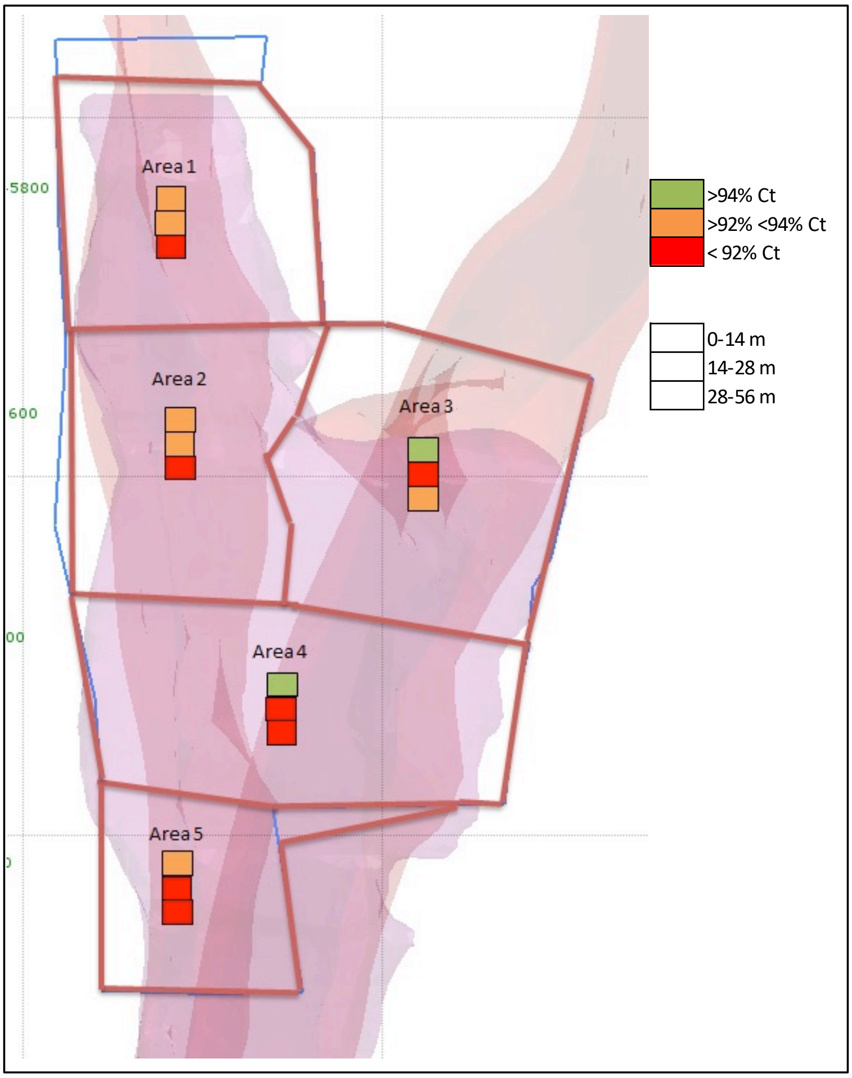


Figure 38: Location and Performance of Area Composites

The average open circuit carbon recovery for the fifteen area composite tests was 87.6% and ranged between 73.2% for the Area 1 (14m to 28m) composite and 93.6% for the Area 3 (28m to 56m) composite.

Since open circuit tests treat the intermediate cleaner tails as final tails, the recoveries are lower compared to closed circuit operation. To determine the closed-circuit performance, four locked cycle flotation tests were completed on the Molo mineralization.

The analysis of the results determined that 66% of the carbon units that reported to the intermediate cleaner tails report to the final concentrate in closed circuit operation.

This factor was applied to the fifteen tests using the area composites to arrive at a closed-circuit carbon recovery Projection of 90.5%.

13.8 Concentrate Upgrading Tests

The average concentrate grade of 92.1% of the area composites has a significant impact on the economics of the Project. Concentrates grading between 94% and 97% total carbon are in higher demand and achieve substantially higher prices compared to a concentrate grading only 90% total carbon. To increase the graphite flotation, concentrate from approximately 92% total carbon to at least 94% total carbon, two upgrading strategies were evaluated.

The first approach applied high temperature drying at 400°C for one hour followed by classification of the dried concentrate on a standard set of sieves. It was postulated that the high temperature drying for an extended period of time could possibly weaken, or break the bonds between the graphite layers and non-sulphide gangue minerals within the intercalated graphite flakes.

Screening the product could then result in upgrading of the coarser graphite flakes if the gangue minerals are liberated in the drying process and report to the smaller size fractions.

This upgrading approach failed to improve concentrate grades and a cleaner flotation test conducted on the dried concentrate did not produce further grade improvements.

Optical mineralogy that was conducted on the dried concentrate confirmed the existence of intercalated graphite, which led to the rejection of this upgrading strategy.

The second upgrading strategy evaluated a series of different sized grinding media and grinding mills, as well as sodium silicate as a gangue depressant. A combined concentrate from the Phase II variability program was homogenized and split into equal test charges that were then subjected to five different upgrading conditions.

The most promising results were achieved using an attrition scrubber with 1 mm ceramic media and a stirred media mill with 6 mm steel media.

Ten additional tests were carried out using the attrition mill and attrition scrubber. A weighted combined concentrate of all fifteen area composite tests was generated for those tests, which was a good representation of the average mineral resource.

The variables that were modified in the ten tests were the grinding times and the use of sodium silicate.

While the test using the 1 mm ceramic media in an attrition scrubber produced good and combined concentrates grades of more than 96% total carbon, the flake size degradation was significantly higher compared to the tests using the stirred media mill. This is evidenced in Figure 39 and Figure 40 and which depict the mass recovery into the size fractions of the final concentrates of tests conducted with the stirred media mill and attrition scrubber, respectively. To quantify the degree of flake degradation, the charts also include the data for the feed sample prior to milling or scrubbing.

The degradation of the flakes larger than 65 mesh was less pronounced for the stirred media mill and even at the longest grind time the mass recovery into the +48-mesh size fraction was still 17.1%. In contrast, the shortest grind time in the attrition scrubber reduced the mass recovery into the +48 mesh concentrate to 14.3%. The shortest grind time in the stirred media mill reduced the mass of the +48 mesh and -48/+65 mesh concentrate by only 2.6% and 2.7%, respectively.

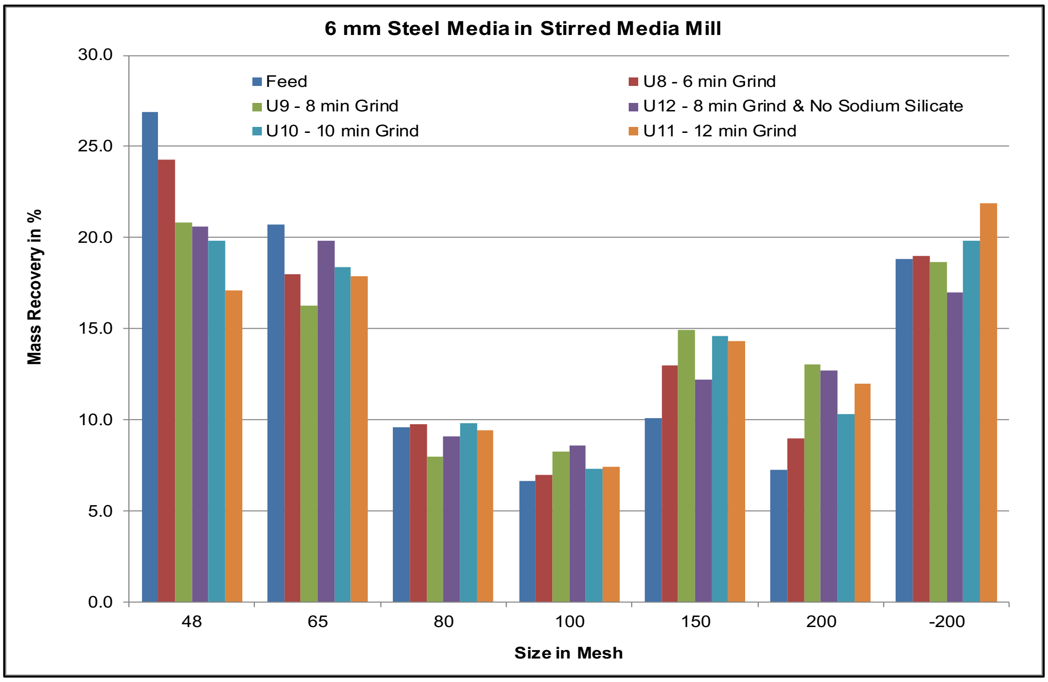


Figure 39: Stirred Media Mill Size Fraction Analysis – Mass Recovery

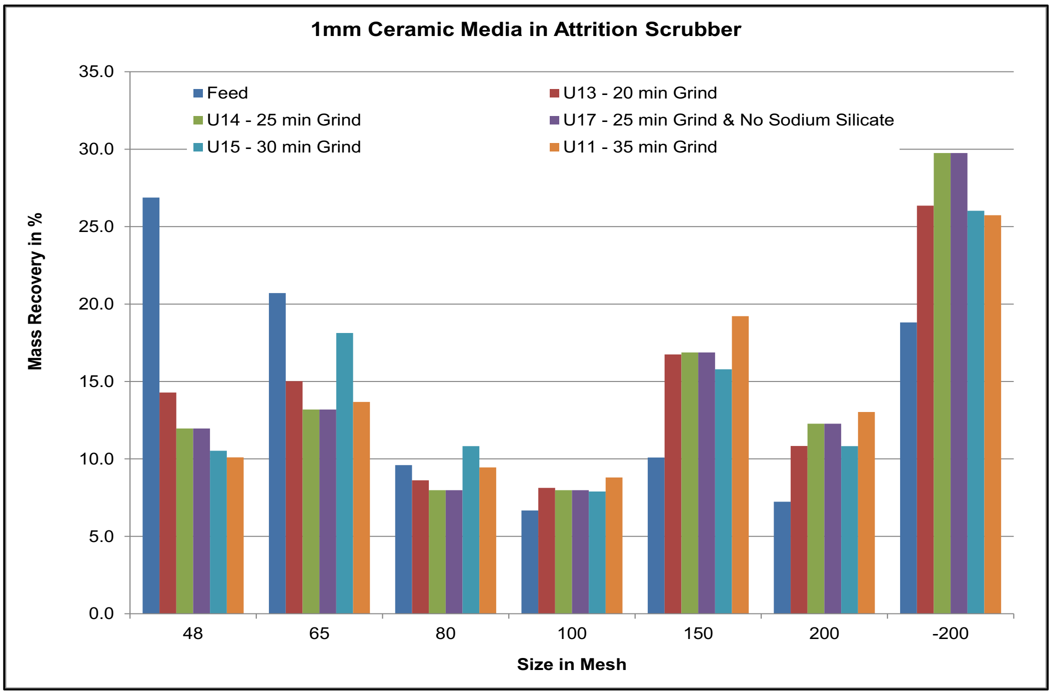


Figure 40: Attrition Scrubber Size Fraction Analysis – Mass Recovery

The total carbon grades into the size fractions of the combined concentrate for the five tests using the stirred media mill are presented in Figure 41. The shortest grind time of 6 minutes produced concentrate grades of 96.9% to 98.1% total carbon for the various size fractions. Even if the worst-case scenario of the relative measurement error of 1.4% associated with the total carbon analysis by LECO SC_632 is applied, the results are consistently above the minimum grade target of 95% total carbon.

Because the flake degradation increased with longer grinding times without a clear improvement in the concentrate grades, the test with the shortest grind time was deemed the most successful one.

It should be noted that all size fractions of the concentrates of the five upgrading tests using the stirred media mill yielded at least 96.5% total carbon, which attests the robustness and repeatability of the upgrading approach.

It may be possible to reduce the amount of flake degradation in the large and jumbo flake categories with the addition of a classification stage of the intermediate concentrate at 80 mesh prior to stirred media milling followed by separate cleaning circuits for the screen oversize and undersize fractions. This approach allows to tailor the final graphite concentrate grade distribution to specific market demands.

A comparison of the results of tests U9 (sodium silicate at 40 kg/t of concentrate) and U12 (no sodium silicate) reveals that the gangue depressant only increased the combined concentrate grade by 0.2% from 97.1% to 97.3% total carbon. Since this grade improvement is statistically insignificant, the addition of sodium silicate cannot be justified for inclusion in the upgrading circuit.

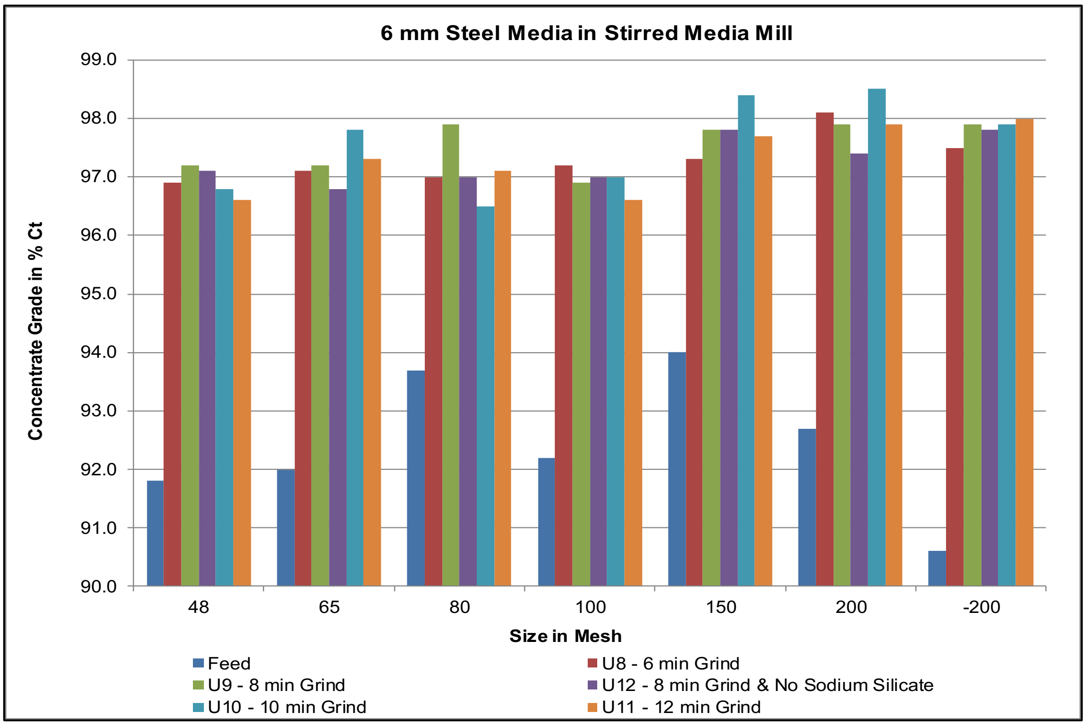


Figure 41: Concentrate Grades of Size Fractions – Stirred Media Mill

The complete Molo flowsheet with the attrition circuit is depicted in Figure 42.

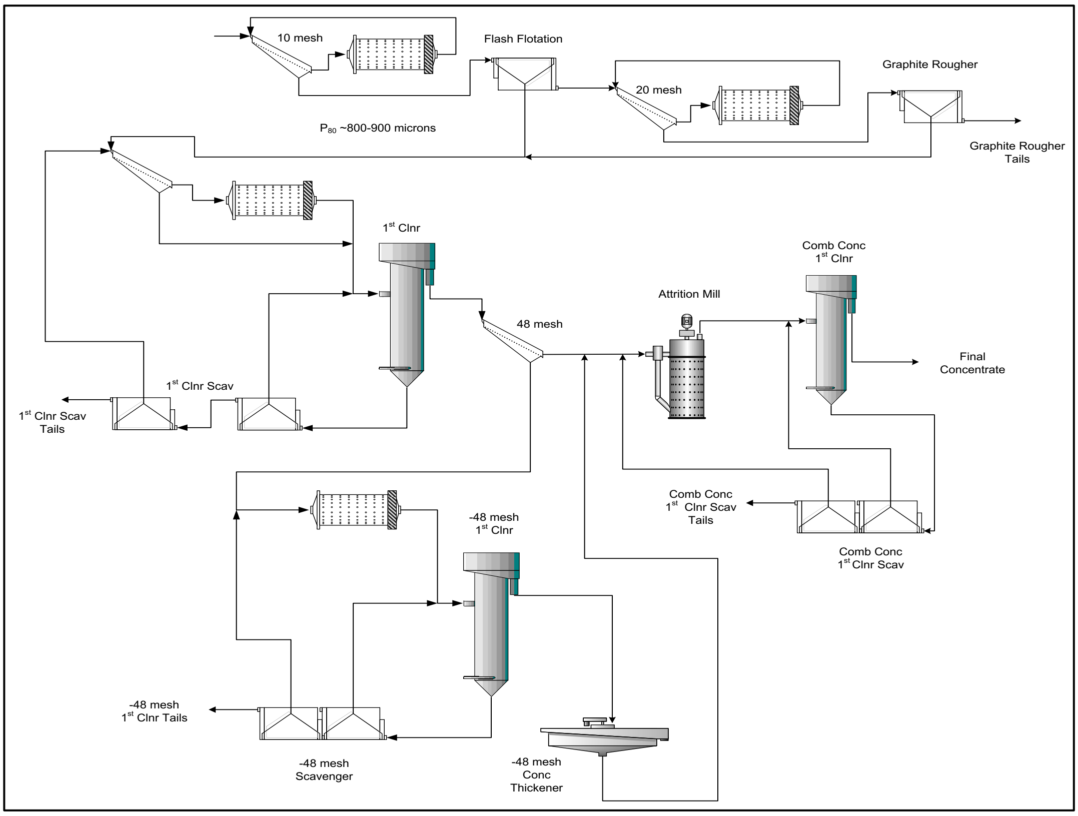


Figure 42: Molo Flowsheet Including Stirred Media Mill Circuit

Since the concentrate used in the upgrading tests was a weighted composite of all area's composite concentrates, the flake size distribution and concentrate grade of test U8 was considered most representative of the average plant product in the first several years of mining operation.

The mass recovery into the various size fractions and associated concentrate grades are presented in Table 26.

The open circuit carbon recovery into the final concentrate of test U8 was 97%. Although carbon recovery will likely increase in closed circuit operation, this conservative number was applied to the circuit carbon recovery of 90.5% prior to upgrading.

As a result, the combined carbon recovery for the main process flowsheet and the upgrading circuit is 87.8%.

Table 26: Mass Recovery and Total Carbon Grades of Size Fractions of Final Concentrate

Product Mesh	Microns	Mass (%)	Grade (% Total Carbon)
+48	+297	24.3	96.9
-48/+65	-297/+210	18.0	97.1
-65/+80	-210/+177	9.8	97.0
-80/+100	-177/+149	7.0	97.2
-100/+150	-149/+106	13.0	97.3
-150/+200	-106+74	9.0	98.1
-200	74	19.0	97.5
TOTAL		100.0	97.2

13.8.1 Optical Mineralogy of Upgraded Concentrate

The seven size fractions of the most successful upgrading test U8 using the stirred media mill were submitted for optical mineralogy. All samples displayed similar mineralogical characteristics.

Non-sulphide gangue (NSG) minerals were generally fine grained (<20 microns to 500 microns). NSG occurred as minor liberated grains only in the +48 mesh, and sporadically in some of the other fractions.

The bulk of the NSG occurred inter-layered with graphite grains. They were developed along the long axis of the graphite particles and are of varied width.

A photomicrograph of graphite flakes in the +48-mesh size fraction of the U-8 third cleaner concentrate is depicted in Figure 43 to illustrate the inter-layering. The image shows graphite (red arrows) that is largely liberated in the sample. However, non-sulphide gangue minerals (green arrows) are mainly inter-layered within graphite.

While the inter-layering has not been eliminated in the upgrading stage, the frequency has been reduced significantly, thus resulting in the mean grade improvement of approximately 5% total carbon.

It is postulated that the coarser intercalation between graphite and NSG minerals is separated efficiently in the stirred media mill, but that the thinner layers between graphite and NSG minerals are more difficult to segregate. However, given the more than adequate concentrate grade of 97.2% total carbon in the U8 test, eliminating all inter-layering is not required.

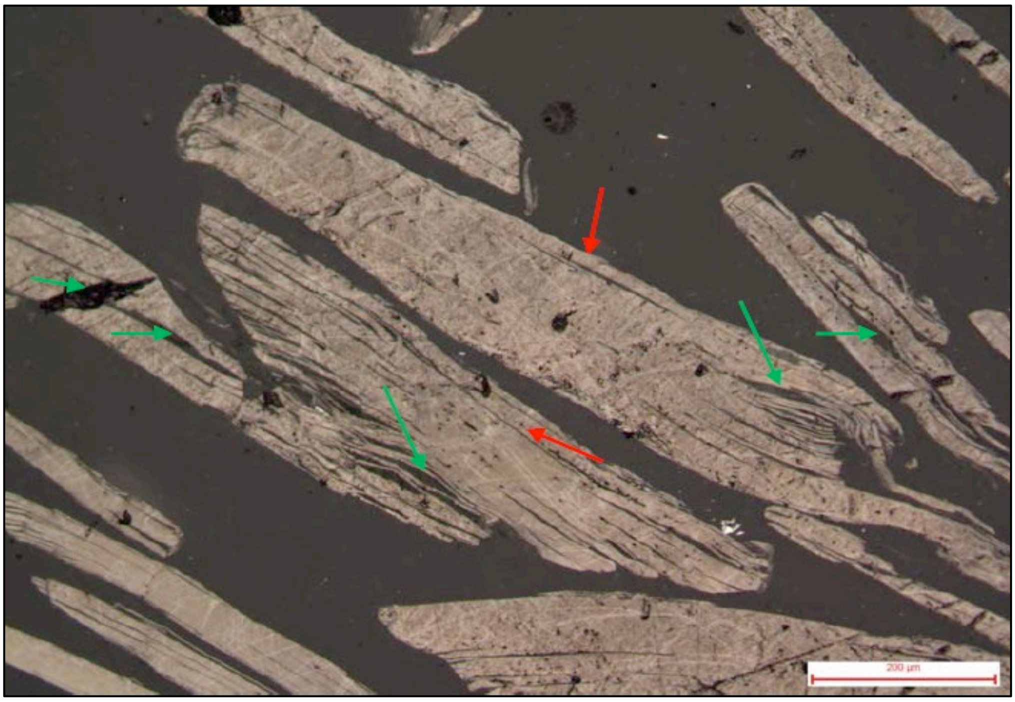


Figure 43: Optical Photomicrographs (Ppl) From The U-8 SFA 3rd ClN CONC +48 Mesh

13.9 Total Carbon and Graphitic Carbon Assay Methods – SGS

Carbon occurs as organic carbon, carbonate carbon and graphitic carbon. The three types of carbon combined represent the total carbon content of a sample.

The total carbon content of a sample is determined by combusting a pulverized sample followed by infrared detection on LECO instrumentation. Lower grade samples are analysed with a LECO 844, while samples higher than 30% carbon are analysed for carbon on the SC632 instrument. Both instruments use high temperature combustion followed by infrared detection of CO2.

The graphitic carbon content of a sample is determined in a three step process using coulometric analysis. The pulverized sample is roasted in an oven at 500°C for 15 minutes to remove any organic carbon. The carbonate carbon is determined by subjecting one aliquot of the roasted samples to coulometric analysis. A second aliquot is used to determine total carbon using a tube furnace. The graphitic carbon is then calculated by the difference between the total carbon and carbonate carbon.

13.10 Additional Testing

The physical properties of graphite are very different to most other commodities because of its particle shape and density. Consequently, it is essential that all unit operations for a proposed graphite processing plant are evaluated in laboratory, or pilot scale trials to obtain robust data for the process design criteria.

Graphite concentrate that was generated in the 2013 pilot plant campaign was shipped to various equipment vendors to evaluate dewatering, drying, and screening applications. These unit operations are required to produce a final saleable product from the initial graphite flotation concentrate.

Further, dewatering tests were carried out on the combined tailings from the pilot plant campaign as poor settling properties of the fine particles in the tailings were observed in laboratory and pilot scale testing.

13.10.1 Thickening

Two equipment Vendors conducted thickening test work on concentrate and tailings samples that were generated in the pilot plant campaign.

Both Vendors conducted static settling and dynamic tests to identify a suitable flocculant and to establish process parameters to achieve a high thickener underflow density and clear overflow.

Vendor A quantified a solids loading rate of 0.25 t/m^2/h for the concentrate thickener yielding an underflow density of 36% solids at a flocculant dosage of 5 ppm Magnafloc 919. The overflow contained 230 ppm solids. The tailings required the addition of a coagulant to achieve satisfactory overflow clarity of less than 100 ppm. The reagent regime consisted of 5 ppm Magnafloc 1011 and 500 ppm Magnafloc 370. An underflow density of 50% solids with an overflow containing 70 ppm solids was achieved at solids loading rate of 0.75 t/m^2/h.

Vendor B quantified the solids loading rate for the concentrate thickener at 0.64 t/m^2/h at a flocculant SNF 905 VHM dosage of 20 g/t. The solids loading rate for the graphite tails was 0.46 t/m^2/h at a flocculant SNF 934 VHM dosage of 140 g/t. The concentrate and tailings thickener underflow solids concentration at 2-hour retention time were 41% and 47% respectively. The overflow clarity for the graphite concentrate was clear, at less than 100 ppm, while the graphite tailings contained a solids concentration of 3,500 ppm.

Vendor A required a large dosage of coagulant Magnafloc 370 to produce an overflow clarity for the graphite tailings thickener application of less than 100 ppm suspended solids, while Vendor B failed to generate an acceptable graphite thickener tails overflow clarity. Two reagent suppliers, supplier A and supplier B, were contracted to carry out a more comprehensive reagent screening to evaluate a reagent regime requiring lower dosages.

Supplier A recommended the use of approximately 125 g/t of flocculant Magnafloc 24, 155, 1011, or 919 in conjunction with 100 ppm– 150 ppm of coagulant Magnafloc 1707 to achieve the desired overflow clarity of less than 100 ppm suspended solids in the graphite tailings thickener.

Supplier B did not develop a reagent regime that achieved dosages lower than the ones recommended by equipment Vendor one.

13.10.2 Filtration Tests

Two suppliers conducted filtration test work on concentrate and tailings samples that were generated in the pilot plant campaign.

The first vendor conducted bench scale testing to evaluate filter cloth selection, filter cake thickness, filtration rate, moisture content of the cake, and cake handling characteristics to achieve 15% to 20% w/w moisture in the concentrate cake.

The tests conducted by the first Vendor produced concentrate filter cakes with cake moisture content between 11.0% w/w and 20.5% w/w at filtration rates of 179 kg to 417 kg DS/m^2/h. Filtration tests on the tailings produced filter a cake moisture content between 12.7% and 17.9% w/w at filtration rates of 92 kgs to 218 kg Ds/m^2/h.

Vacuum filtration tests conducted by the second vendor produced a concentrate cake moisture content of 23% w/w at a filtration rate of 327 kg Ds/m^2/h.

Pressure filtration tests on the concentrate produced a cake moisture content of 23.2% w/w. The vacuum filtration properties of the tailings were poor yielding a cake moisture content of 32% w/w at a filtration rate of 41 kg Ds/m^2/h.

13.10.3 Concentrate Drying

Drying tests were conducted using a rotary dryer and a fluid bed dryer.

While the rotary dryer did not operate well treating the as-received graphite concentrate with a moisture content of 32% to 39% w/w, good performance was obtained when back-mixing some of the dried concentrate to adjust the feed moisture content to 26% w/w.

Since the filtration tests conducted by both vendors produced filter cakes with a lower moisture content than 26% w/w, the rotary dryer is a suitable drying technology to achieve a product moisture content of less than 0.5% w/w.

The fluid bed dryer work failed to produce results.

13.10.4 Wet and Dry Screening Tests

Wet and dry screening tests were carried out at one Vendor to evaluate screening applications on intermediate and final graphite concentrates and to determine if screens could be employed in a dewatering application.

Classification of the dried graphite concentrate was performed at 50 mesh, 80 mesh and 200 mesh using dried concentrate that was generated during the rotary dryer tests.

The dry screening tests suggested that classification can be carried out at a rate of 1.0 t/h per meter of screen width.

Tests on a wet graphite concentrate to evaluate the classification of the intermediate concentrate at 80 mesh and 140 mesh yielded screening rates of 2.0 t/h to 2.2 t/h per meter of screen width.

Dewatering tests on the -140 mesh material was carried out on a 270 mesh screen deck. The mass recovery into the screen oversize was 63.4% at a moisture content of 49.7% w/w.

13.10.5 Additional comments

The tails stream from the upgrading, or attrition circuit is being pumped out to final tails, or discard.

The feed to the Attrition circuit is of significantly high grade, (essentially already a product, just not the right final grade) and the attrition circuit serves only to upgrade the product to a higher grade. So, it follows that the tailing stream from this circuit should also still be of high grade enough to warrant that it be recycled back into the circuit.

This stream should not report to final tailings, instead it will be directed to the polishing mill discharge to be treated in the primary cleaning circuit.

Since the FEED study completion some improvements have been made. These will be incorporated into the detailed engineering ahead of construction. The changes are not significant, but will improve product quality.

14 MINERAL RESOURCE ESTIMATES

This Mineral Resource Statement was produced by CCIC in September 2014. The approach and methodologies applied in this resource estimation are in accordance with international resource reporting guidelines, including NI 43-101. Leapfrog™ software was used to construct volumetric solids for the zones of mineralisation. The three-dimensional resource modelling, as well as the geostatistical techniques for grade estimation was undertaken using Datamine™. The key assumptions and methodologies used for this resource estimate are fully outlined below.

Note that no further work was done for this study in amending the MRE.

14.1 Geological Database

14.1.1 Topography

A three dimensional ("3D") Digital Elevation Model ("DEM") of the topography was supplied by the Company as 0.5 m contours in ascii format. These contours were generated from an airborne survey using the SenseFly drones, in 2014. Collar elevations from trenches and drill holes have been resurveyed using a differential GPS and incorporated into the topography. The topography is flat lying, with the highest elevation in the north-western side and dipping gently to the south (Figure 44). Elevations within the area of study range from 570 mamsl to 543 mamsl, with an average gradient of 2.0^{o}. The high grade domain on the western side creates a ridge up to 2.0m in elevation.

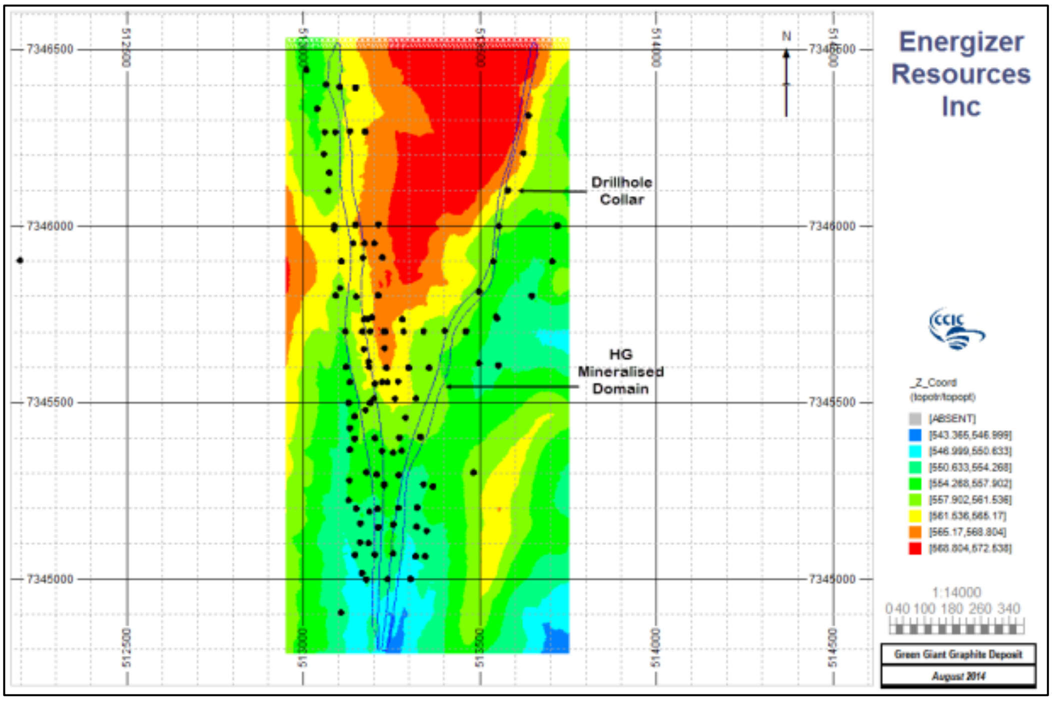


Figure 44: Topographic Contours; Elevations In MAMSL

14.1.2 Drill Holes

This MRE is based on 80 drill holes (total 11,660m) and 35 trenches (total 8,492m) drilled by the Company. Drill spacing varies from 100m * 100m in poorly informed areas to 50m * 50m in well informed areas. Figure 45 illustrates a plan view of the drill holes and trenches, coloured on % C grades. Drill holes are orientated approximately 45° to the east. The database containing drill hole and trench information was supplied by the Company in a Microsoft Access format. Logging codes used for lithological modelling are summarised in Table 27.

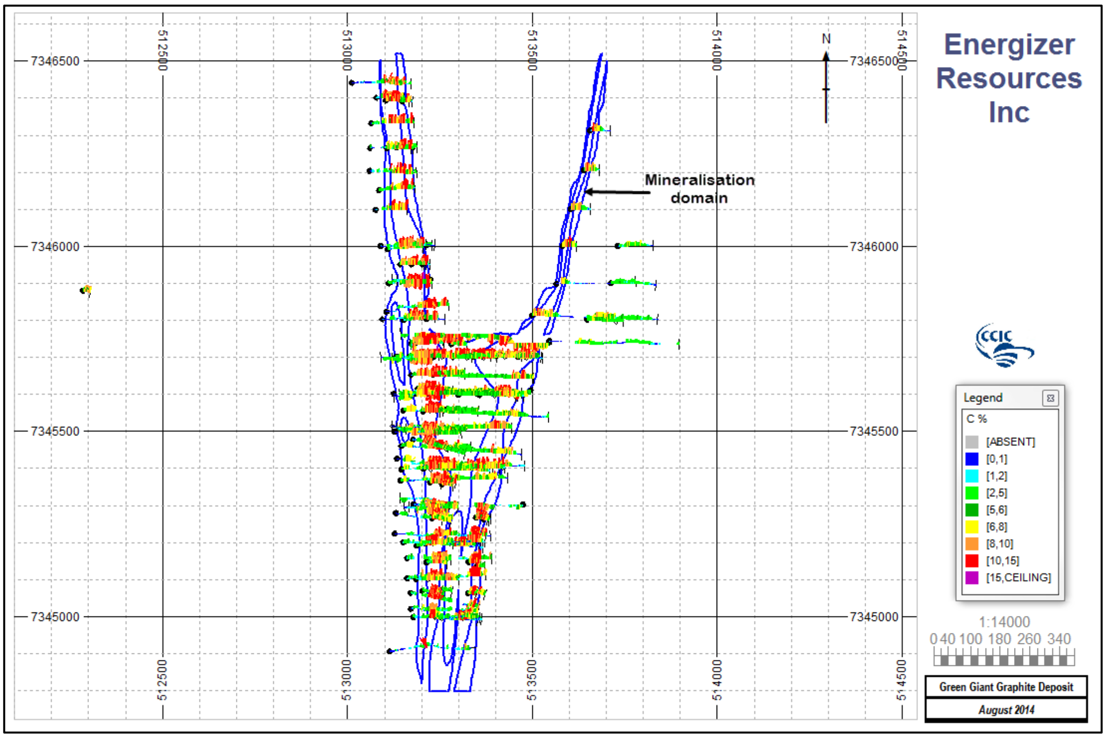


Figure 45: Drill Hole Posting Plan Coloured on C (%) Grades.

Table 27: Summary of Lithological Codes Used

Logging Code	Description
Gp Gn	Graphite Gneiss
Gt Gn	Garnet Gneiss
Mb	Marble
SAPR	Saprolite
PEG	Pegmatite
OVBN	Overburden

Spatial and statistical comparisons of % C between drill holes and trenches are presented in Figure 46 and Figure 47. There is good spatial and statistical correlation between the two datasets.

Overall, trenches show a slight positive bias for the Mean. Reason for this may be because the infilling drilling program focussed on upgrading the higher grade portions of the deposit.

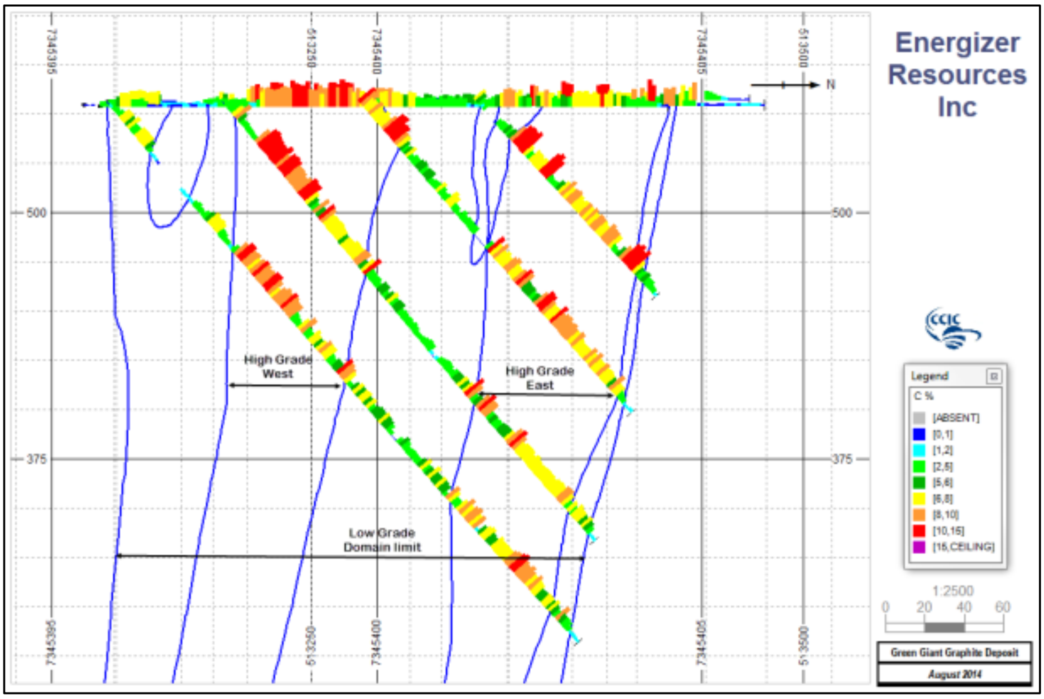


Figure 46: Cross Section Showing the Grade Distribution (C%) Nn the Drill Holes and Trenches

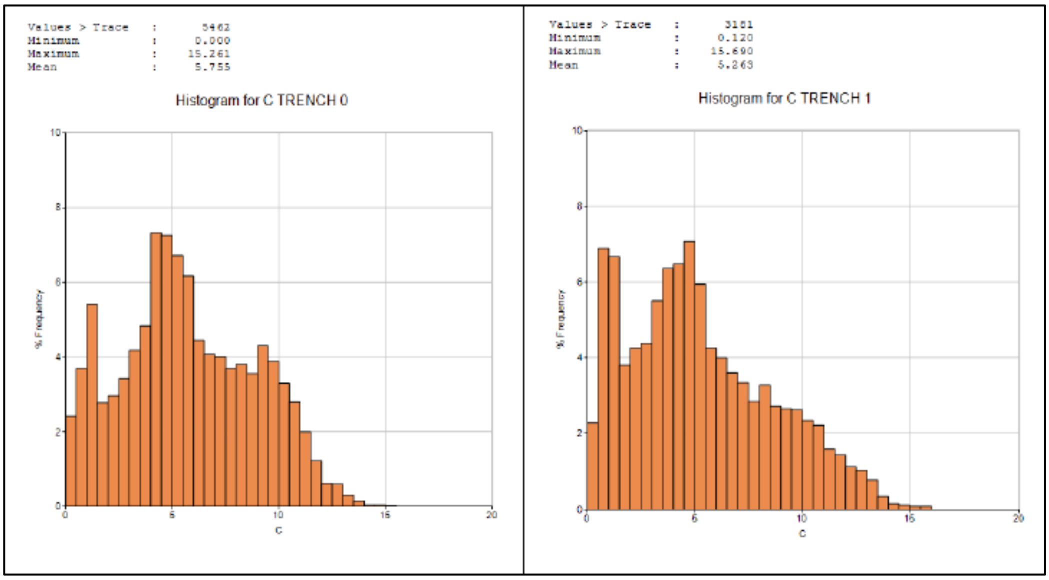


Figure 47: Statistical Comparisons Between Diamond Drill Hole (Dd) and Trench (Th) Samples

14.1.3 Relative Density

A total of 226 RD measurements are contained in the Molo database, 179 of which were for the graphitic gneiss (GpGn). These are presented as a histogram in Figure 48 below. The average RD for all 226 readings is 2.39 t/tm^3. RD values within the GpGn range from 1.59 t/tm^3 to 2.95 t/tm^3, with an average of 2.35 t/tm^3.

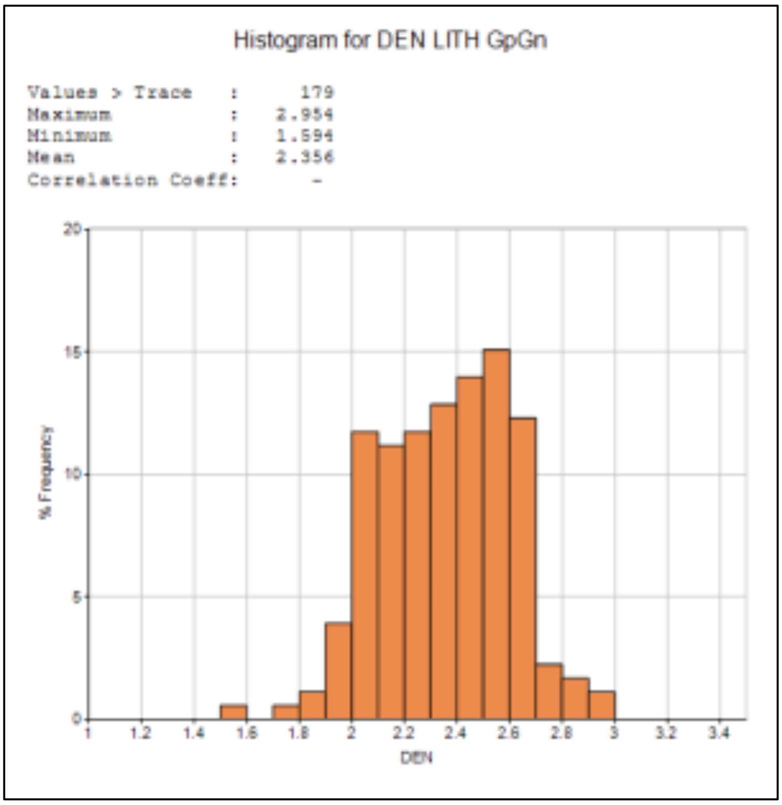


Figure 48: Histogram of Relative Density Readings within the Graphitic Gneiss

14.2 Geological Model on Which the Grade Estimation is Based

14.2.1 Grade Domaining

The deposit is split into three domains of mineralisation, namely:

A “barren” or “un-mineralized” domain. This is a lithological boundary that separates the un-mineralised Garnet Gneiss from the mineralized Graphitic Gneiss. This boundary was treated as a “hard” boundary during grade domaining and estimation.

The mineralized Graphitic Gneiss has been sub-domained into separate “low grade” and “high grade” domains, based on C grade characteristics. Histogram (%) of C grades prior to sub-domaining illustrates a bimodal distribution. A threshold grade of 6% C was therefore used as a guideline to sub-domain the “low’ and “high” grade zones, while maintaining spatial continuity.

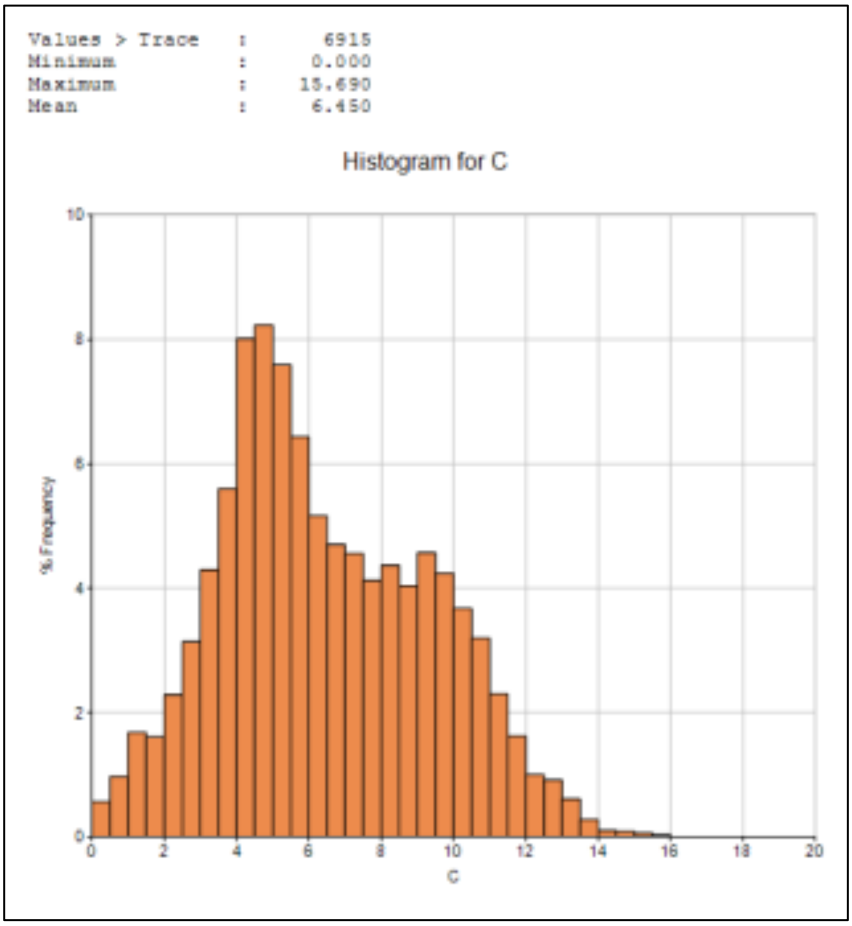


Figure 49: Histogram of C Distribution

14.2.2 Grade Domaining using Leapfrog

Leapfrog is an implicit 3D modelling engine that works of a Radial Basis Function. The modelling methodology differs from the traditional way of deterministically digitising out the zones of interest along section lines, then stitching them together to create a 3D wireframe. Leapfrog is instead based on an algorithm that uses all the data points in 3D space, together with geological constraints and parameters to automatically generate volumes of interest. The benefits of using Leapfrog are that:

Interpretations are not limited to drill holes along a section line. Incorporating drill holes from neighbouring section lines makes the model a full 3D interpretation, ensuring good correlation between section lines.

The algorithm can generate very complex forms, resulting in more efficient domaining.

Because the “low” and “high grade” domains are generated concurrently, there are no overlaps, or protrusions between domains.

The contacts for the three domains were flagged using the “Interval Selection Tool” in Leapfrog, which allows the user to interactively determine the intervals that are to be included, or excluded, from the different domains.

The mineralised domain is composed of Graphitic Gneiss, within which occur two lenses of barren Garnet Gneiss. The mineralised domain was further subdivided into a "low grade" domain and two "high grade" lodes. The "high" grade lodes are referred to as "high grade east" and "high grade west" domains. An isometric, and a section view, are illustrated in Figure 50 and Figure 51 respectively. The mineralisation limits, represented by the graphitic gneiss are shown as pale green, with the barren Garnet Gneiss represented as a white background. Mineralisation strikes approximately north-south, dipping between 75° and 80° to the west. The thickness of the "low grade" zone varies from 60.0 m, where only the "high grade west" domain is developed, to more than 260m in the central portions. "High grade" domains are generally 60m thick, thinning to the south.

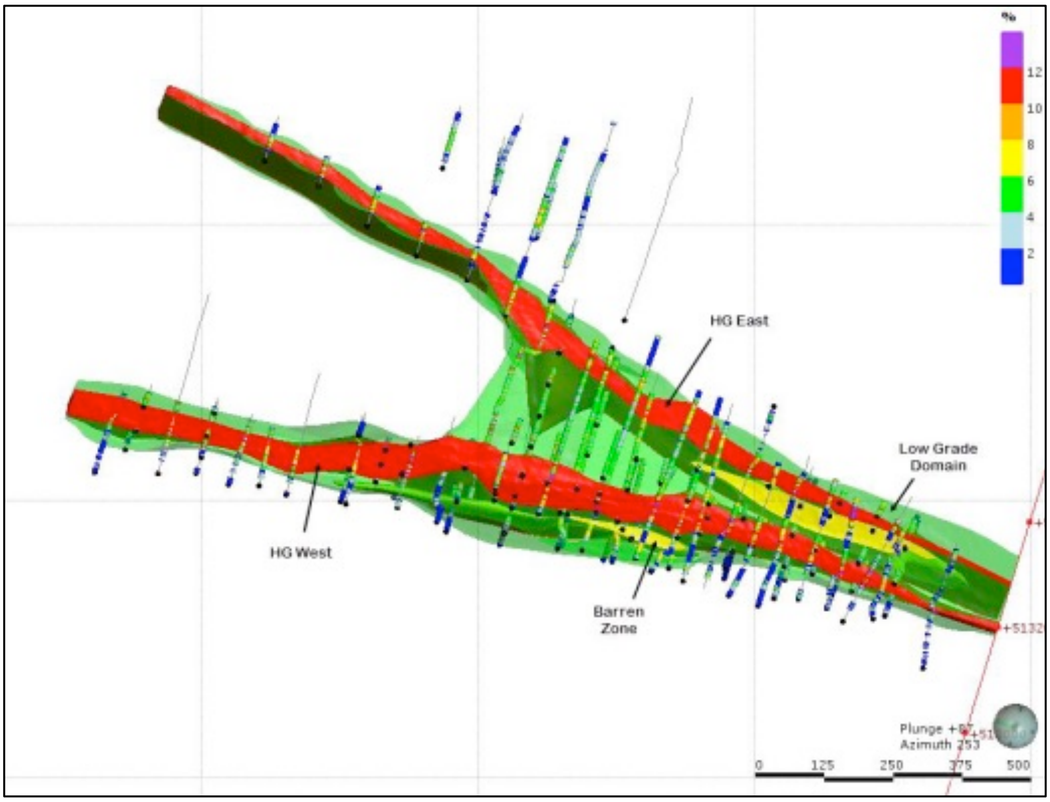


Figure 50: Isometric View Showing "Low" and "High" Grade Domains

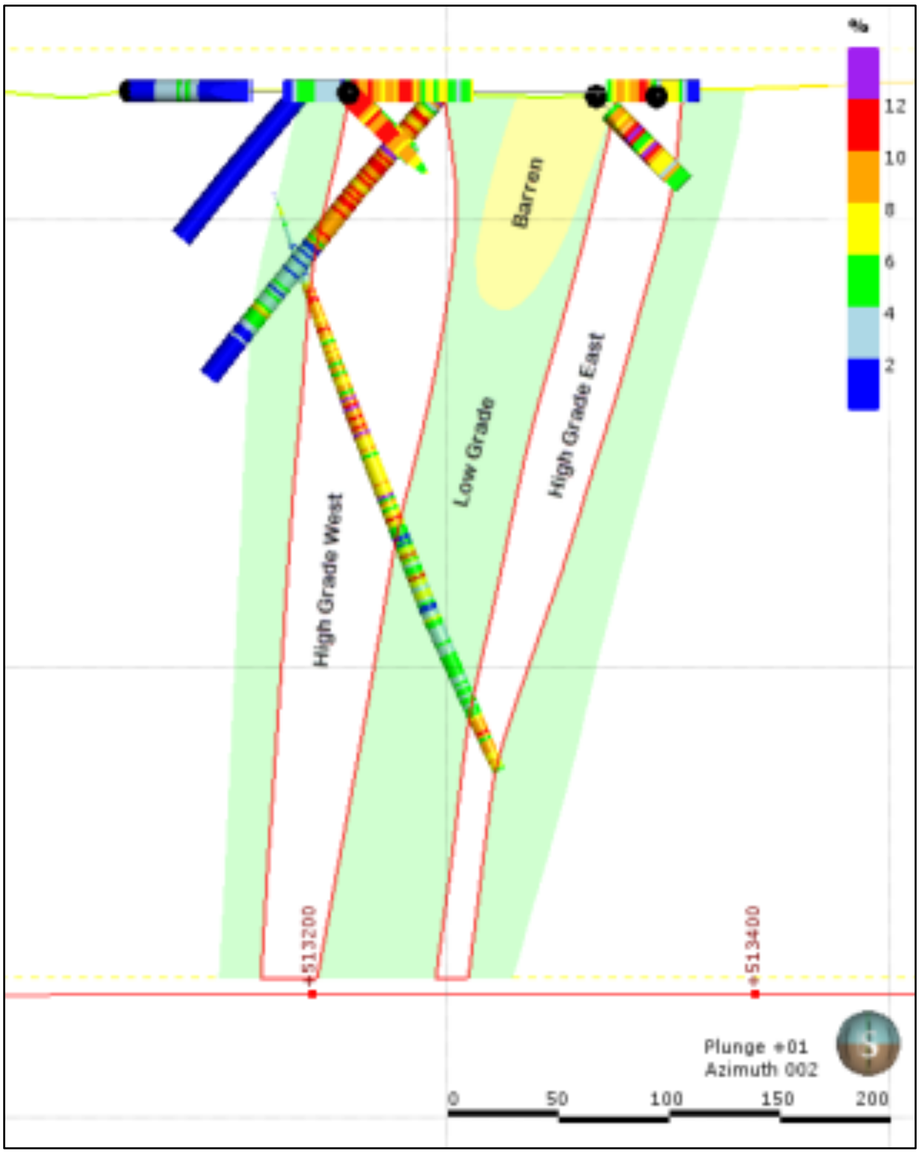


Figure 51: West-East Section Showing the "Low" and "High" Grade Domains

14.2.3 Domaining in Datamine

The envelopes generated in Leapfrog were imported into Datamine for sample and block model flagging. Due to the gradational nature of the boundary between the "low" and "high" grade domains, boundary analysis was undertaken. As illustrated in Figure 52, the contact between the "low" and "high" grade zones was treated as a "soft" boundary. A transition of 5.0m was used to flag samples from the "high" into the "low" grade zones. The boundary was treated as "hard" from "low" into "high" grade.

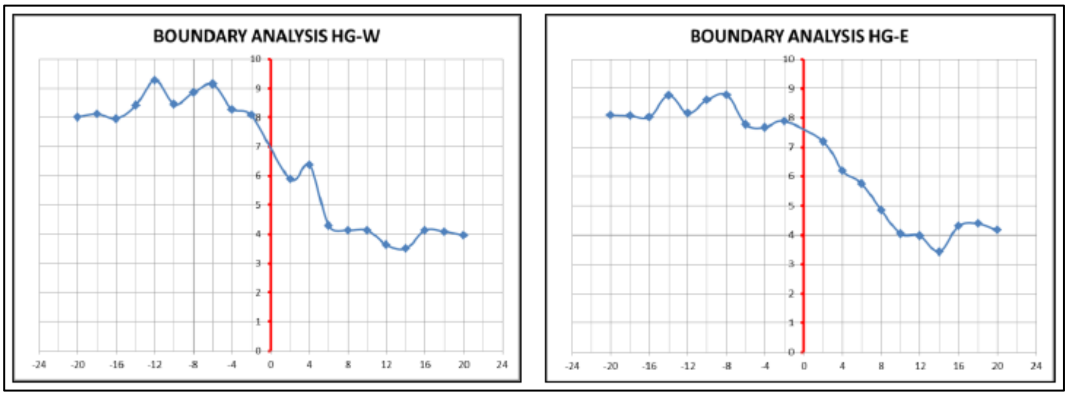


Figure 52: "High" and "Low" Grade Boundary Analysis

Zonal flagging in Datamine uses a field called Kzone to distinguish the different grade domains during geostatistical analysis and estimation (Figure 53).

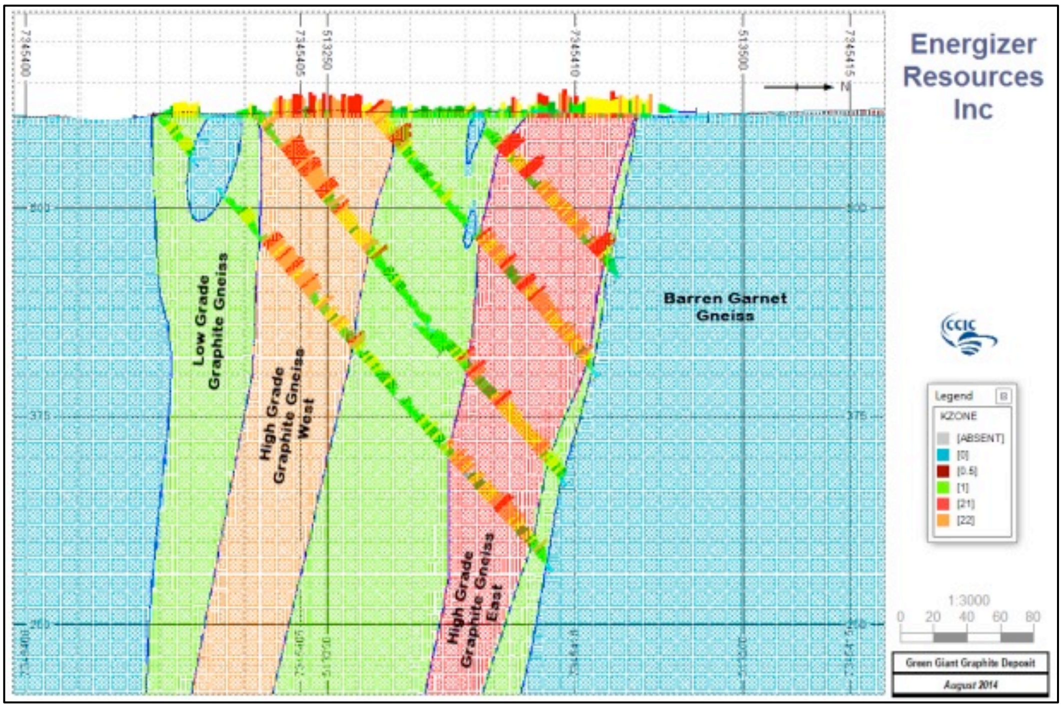


Figure 53: Cross-Section Showing the Kzone Flagging in the Block Model

A summary of the Kzone flagging undertaken in Datamine is provided below in Table 28.

Table 28: Summary of Kzone Flagging in Datamine

Zone Description	Kzone Value	Code Name
Garnet Gneiss Barren Zone	0	WST
Overburden	0.5	OVB
Low Grade Mineralised Zone	1	LG
High Grade Mineralised Zone East	21	HG-E
High Grade Mineralised Zone West	22	HG-W

14.3 Compositing

The predominant sampling interval was either 0.5m, 1.0m, 1.5m, or 2.0m, and hence a composite length of 2.0m was used to include all samples (Figure 54). Compositing used the Kzone to ensure that samples were composited within the different domains. Any samples less than 0.5m after compositing were excluded from the geostatistical analysis and estimations.

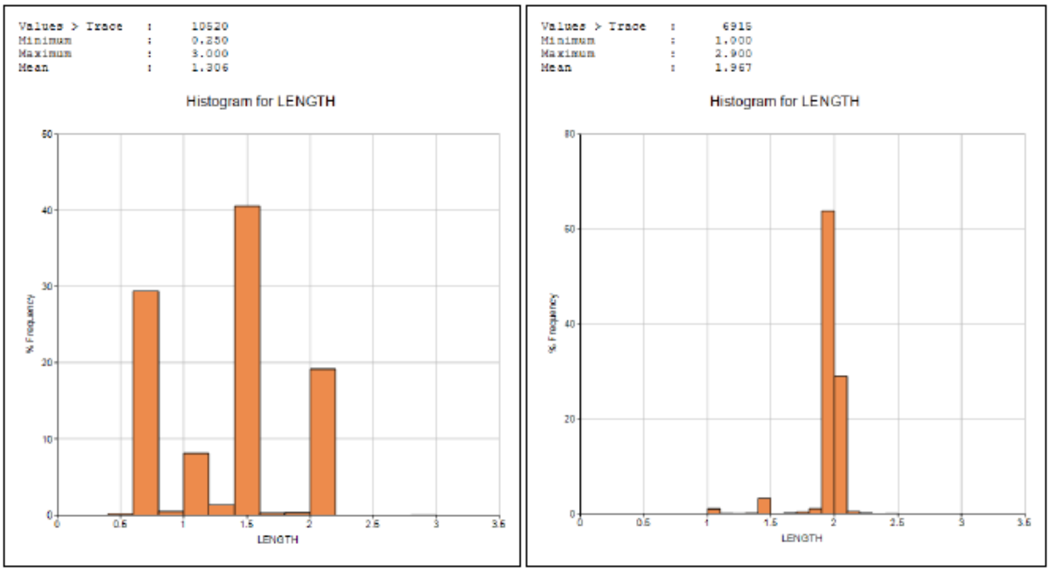


Figure 54: Histogram of Sample Length Prior to and After Compositing

14.3.1 Composited Statistics

The sampling protocol for core intervals deemed to be barren with respect to graphite mineralisation is that they were not to be submitted for analyses. All un-sampled intervals have therefore been set to trace prior to compositing and statistical analysis. A statistical summary for the four grade domains is presented in Table 29 below.

Table 29: Statistical Summary of C% Per Kzone

Kzone	**0**	**1**	**21**	**22**
Field	C	C	C	C
Numtrace	1730	3274	1282	2359
Minimum	0.00	0.00	0.08	0.13
Maximum	15.55	15.31	15.04	15.69
Mean	2.07	4.63	7.72	8.30
Variance	3.10	4.15	5.92	6.97
StandDev	1.76	2.04	2.43	2.64
StandErr	0.04	0.04	0.07	0.05
Skewness	1.88	0.91	-0.06	-0.37
Kurtosis	5.47	2.36	-0.07	-0.09
Logestmn	2.26	5.77	7.93	8.51
CoV	0.05	0.44	0.32	0.32
*StandDev = Standard Deviation *CoV = Co-efficient of Variation				

The mean of the samples for “LG” (Kzone 1) domain is 4.63 % C, with a positive grade tail up to 15.31% C. The “HGE” (Kzone 21) domain has a mean value of 7.72 % C, Co-efficient of Variance (“CoV”) of 0.32 and a maximum of 15.04 % C. The “HGW” (Kzone 22) domain has a mean value of 8.30, CoV of 0.32 and maximum value of 15.69 % C. Histograms of sample distributions for both low grade and high-grade domains are presented in Figure 55, Figure 56 and Figure 57 below. The sample distributions for both the “high grade” domains exhibit very similar statistical characteristics.

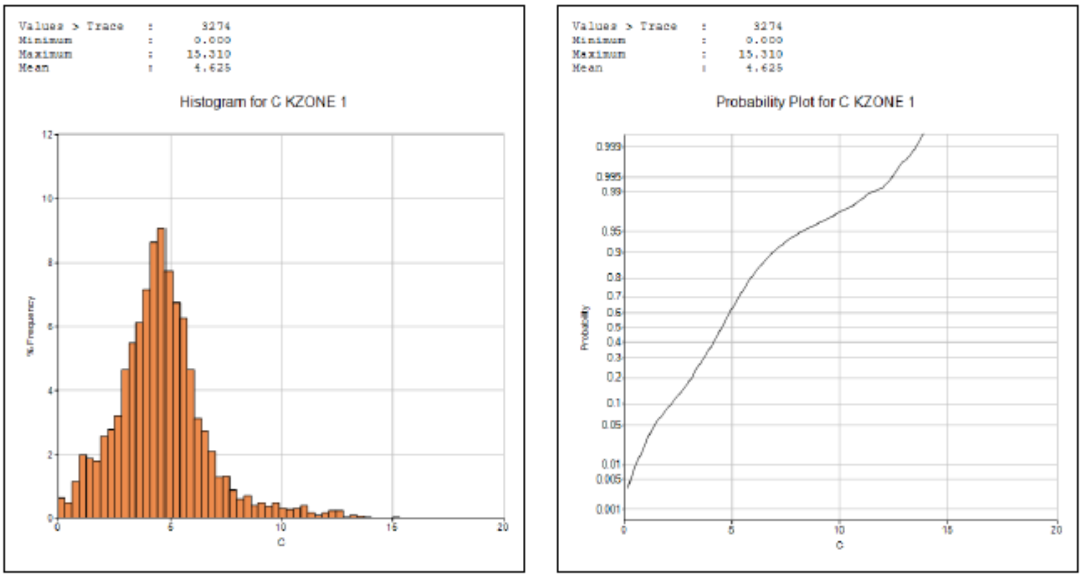


Figure 55: Sample Distributions for the "Low Grade" Domain

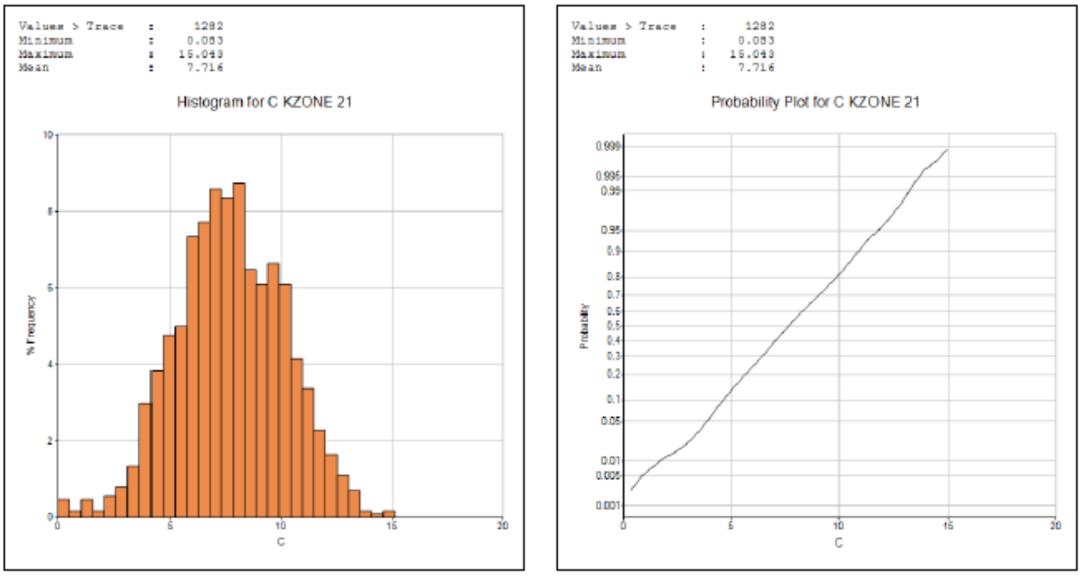


Figure 56: Sample Distributions for the "High Grade East" Domain

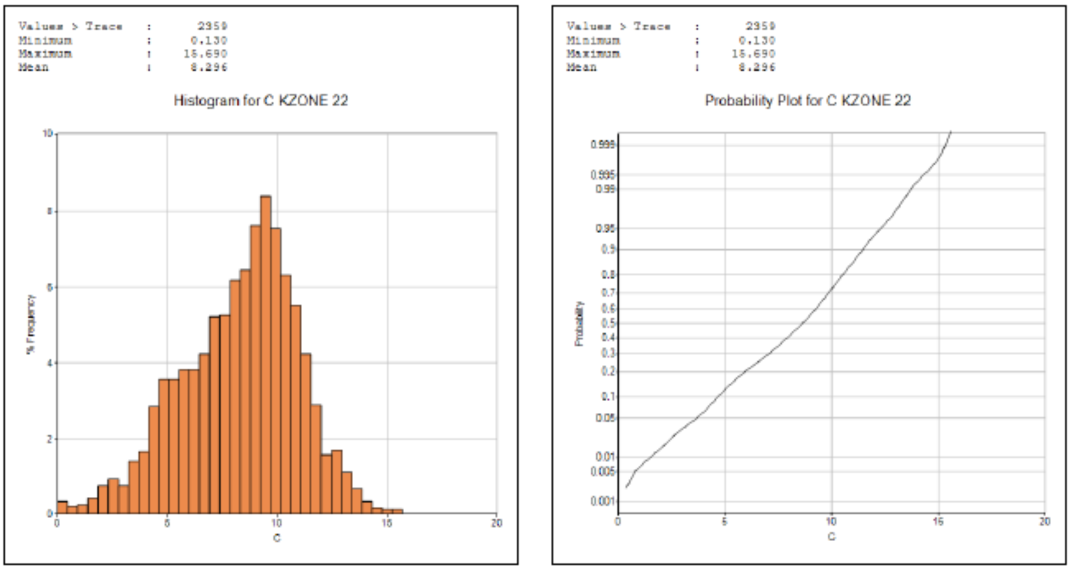


Figure 57: Sample Distributions for the “High Grade West” Domain

14.4 Variography

Variogram analysis and modelling was done using Datamine. Variogram models were generated for % C, for the “low grade”, “high grade” east, and “high grade” west domains. The down hole semi-variograms together with their respective Isotropic models are illustrated in Figure 58, Figure 59 and Figure 60. The nugget for the ‘low grade” domain is 10%, with more than 90% of the spatial variance occurring within the first 120m. The “high grade” east domain has a nugget of 10% and a range that extends to 80.0m. The “High grade” west domain has a nugget of 10% and a range of 55.0m. During the grade estimation the isotropic variogram parameters were used in the strike and dip directions, with the down hole parameters resembling the across strike direction.

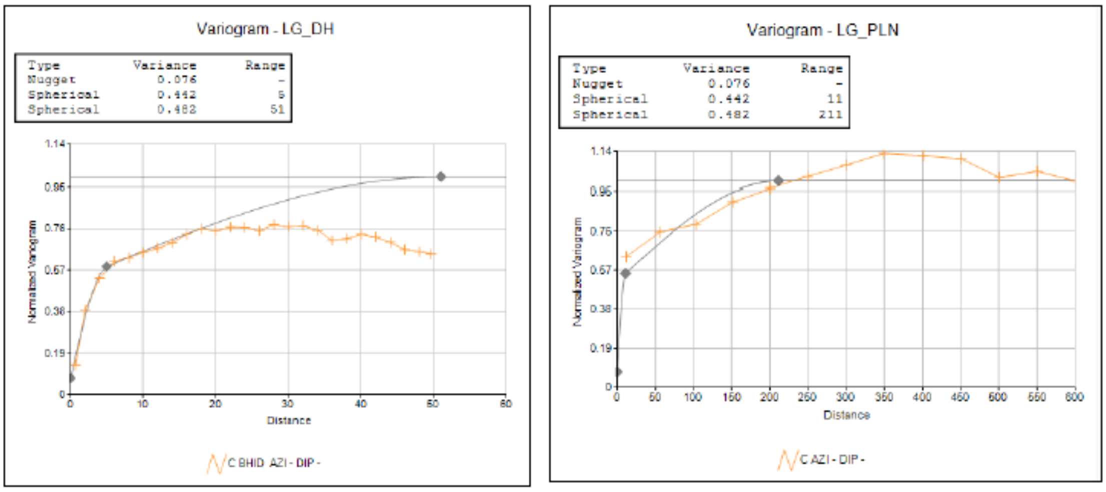


Figure 58: Variogram model for “Low Grade” domain – down hole and isotropic

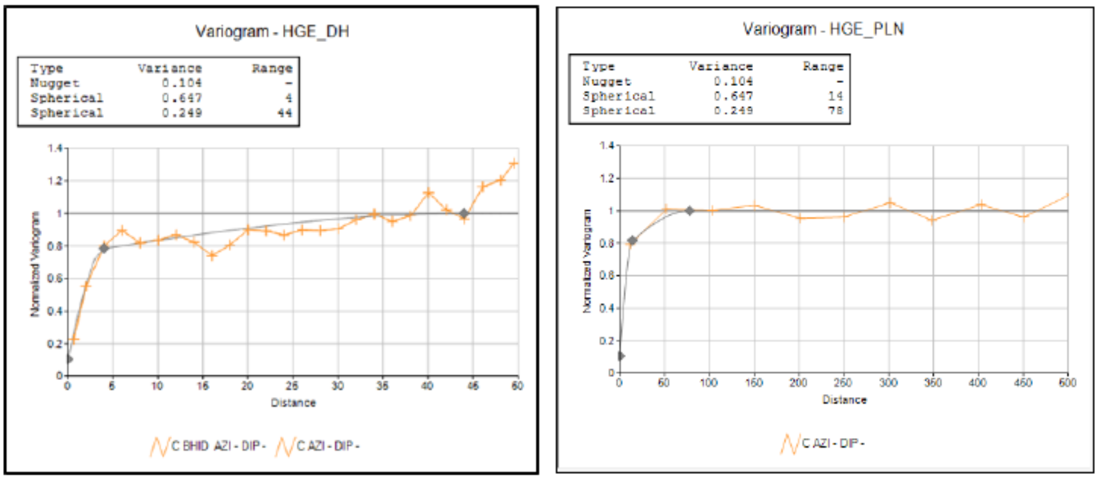


Figure 59: Variogram model for “High Grade East” domain – down hole and isotropic

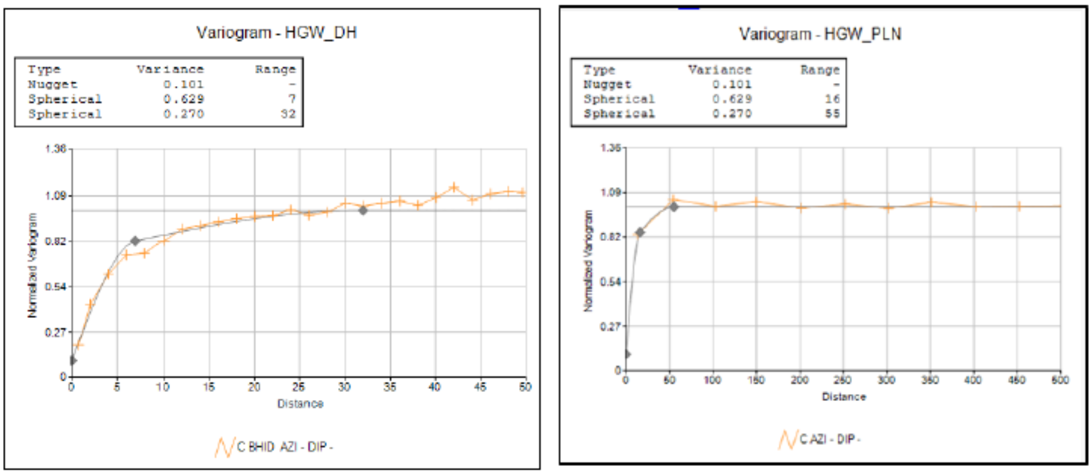


Figure 60: Variogram Model for “High Grade West” Domain – Down Hole and Isotropic

14.5 Top Capping

The top capping strategy considered various criteria to determine the optimum values. These included:

Histograms of sample distributions.

Sample percentiles.

Spatial locations of “outlier” samples.

Validation of model estimates against samples.

A summary of the top capping values that were applied are presented in Table 30 below. All samples that were greater than the top capping value were re-set to the top capping value.

Table 30: Summary of Top Capping Values

Domain	Capping -% C	Percentile	Number of Samples
"LG"	14% C	99.9%	4
"HGE"	15% C	99.9%	2
"HGW"	15% C	99.9%	3

14.6 Grade Estimation

14.6.1 Method

The method of estimations for % C was Ordinary Kriging. Estimations were undertaken using the Estima process in Datamine. Parameters for estimations (i.e. Parent Block size, search distances and the number of samples to be used for an estimate) were optimised using Kriged Neighbourhood Analysis, which is explained in more detail below.

14.6.2 Kriged Neighbourhood Analysis

The aim of Kriged Neighbourhood Analysis is to determine the optimal theoretical search and estimation parameters during Kriging so as to achieve an acceptable Kriging Variance and Slope of Regression ("SOR"), whilst ensuring that none, or a minimal number of samples are assigned negative Kriging Weights. Once this is determined, practicality is taken into account when deciding on the parameters to be used. This optimisation was based on a representative area within the deposit, using the "high grade" west domain, because this is the most prominently mineralised domain. Figure 61 indicates the test location as a block dot with the drill holes coloured on % C values. The following parameters in chronological order were optimised:

Optimum parent cells size for the block model, in X, Y and Z directions.

Optimum search distances in the X, Y and Z directions together with determining the appropriate minimum and maximum number of samples required for a reliable estimate.

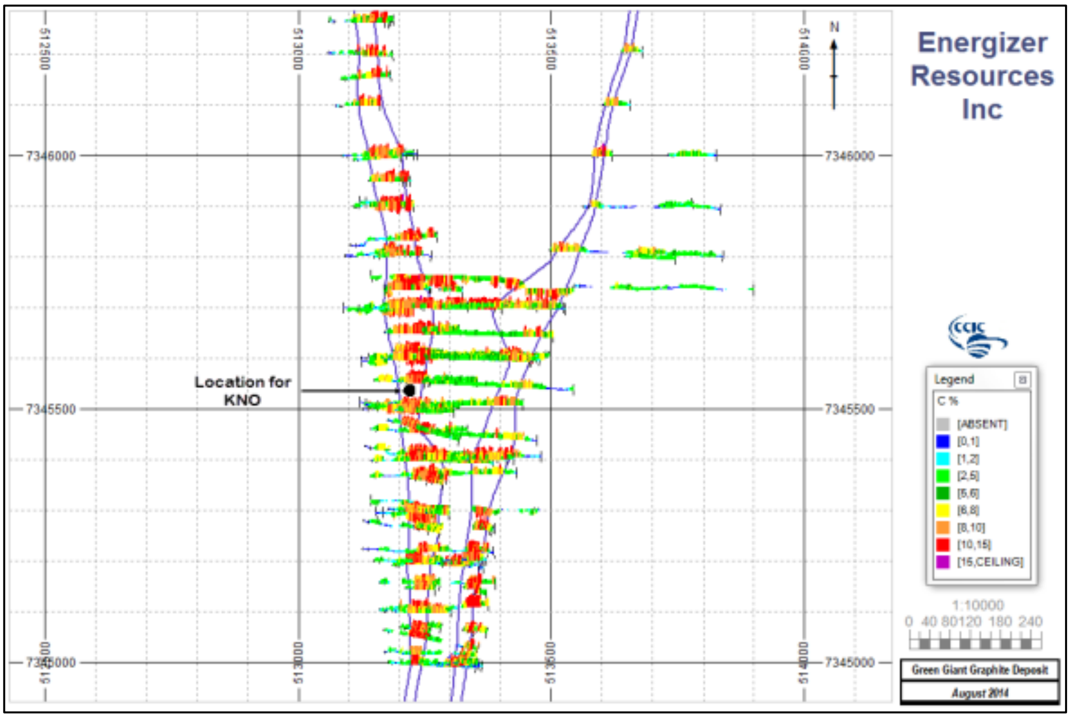


Figure 61: Plan Showing the Location for the Krige Optimizations

During parent cell size optimisation, the approach taken was to use the variogram ranges as a guide to set the search distances, and not to apply any minimum and maximum number of samples to be used. A default parent cell size of 10m*10m*10m was selected using the borehole spacing as a guide. Whilst all parameters remain constant, the parent cell size in the Y direction was set to 5m and then incrementally increased for every estimation. The Estimated Grade (% C), Kriging Variance ("KVAR"), Block Variance ("BVAR"), Kriging Efficiency ("KEF"), SLOR, Number of Samples used ("NUMSAM") and the Percentage of samples with negative Kriging weights ("PCNEGWTH") are recorded for every estimation. These outputs are then plotted against the incremental cell sizes to determine the optimum cell size in the Y direction. The same process is then repeated for the cell sizes in the X direction. Figure 62 compares the plot of the incremental parent cell sizes against the KVAR for the Y direction. The optimum parent cell size, taking practical mining constraints into consideration, was set to 40m*10m*10m in the X, Y and Z directions respectively.

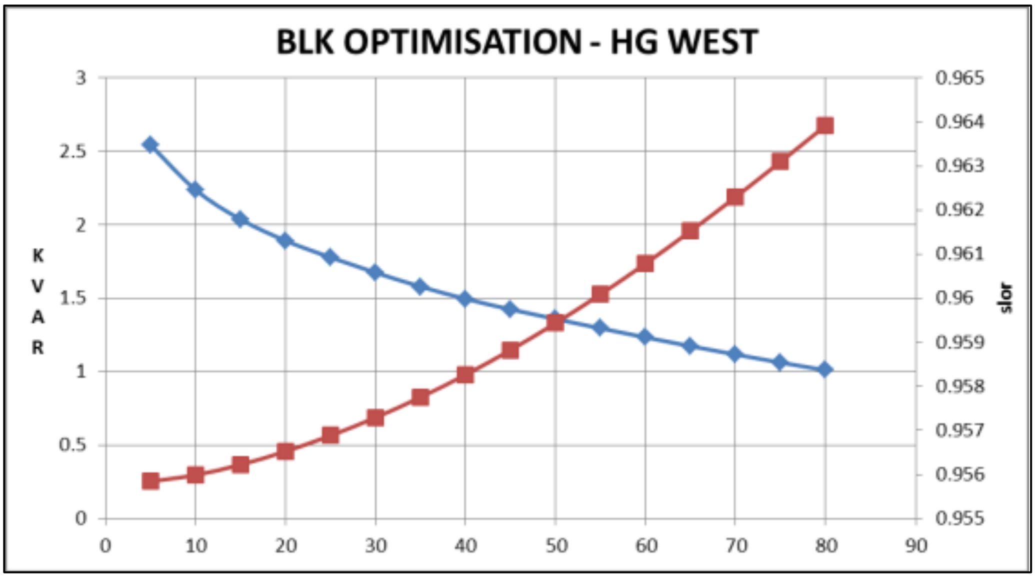


Figure 62: Parent cell size optimizations

During search optimisations, the parent cell size was set to 10m*40m*10m and as described above, search distance in the X direction was incrementally increased. The results of the search optimisation are plotted in Figure 63, which compares both KVAR and SLOR against search distances. Attention is also given the number of samples per estimate and the percentage of samples with negative Kriging weights. The optimum search distance, taking borehole spacing into account, was set as 70m*40m*10m in the X (strike), Y (dip) and Z (across strike) directions respectively. A minimum of 5 and maximum of 100 samples was used per estimate and was limited to five samples per drill hole.

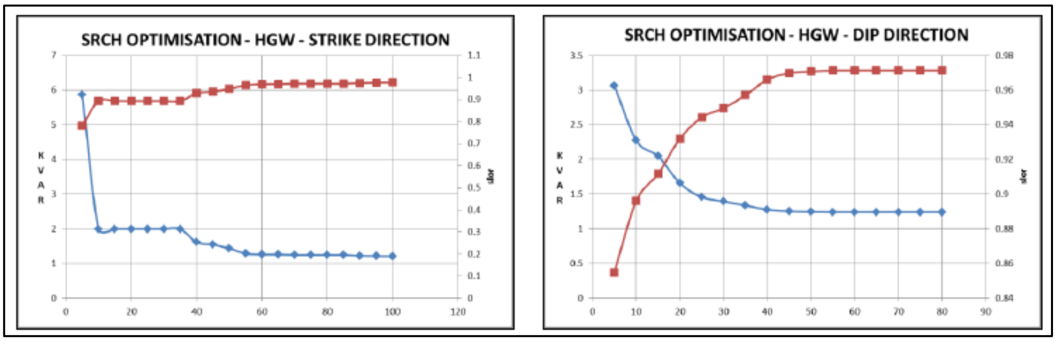


Figure 63: Search Distance Optimizations

14.6.3 Model Construction and Parameters

A block model was constructed using a parent cell size of 10m*40m*10m in the X, Y and Z directions, allowing sub-cell splitting to ensure that the volumes of the graphite mineralised zones were represented. Zonal control was applied during grade estimation with each grade domain in the block model assigned a unique Kzone number, as described under Item 14.2.1 (Grade Domaining) above. A section illustrating the block model colour coded by Kzone is shown in Figure 64. “Waste” domains have been assigned and estimated for indicative purposes only and have not been reported in the mineral resource statement.

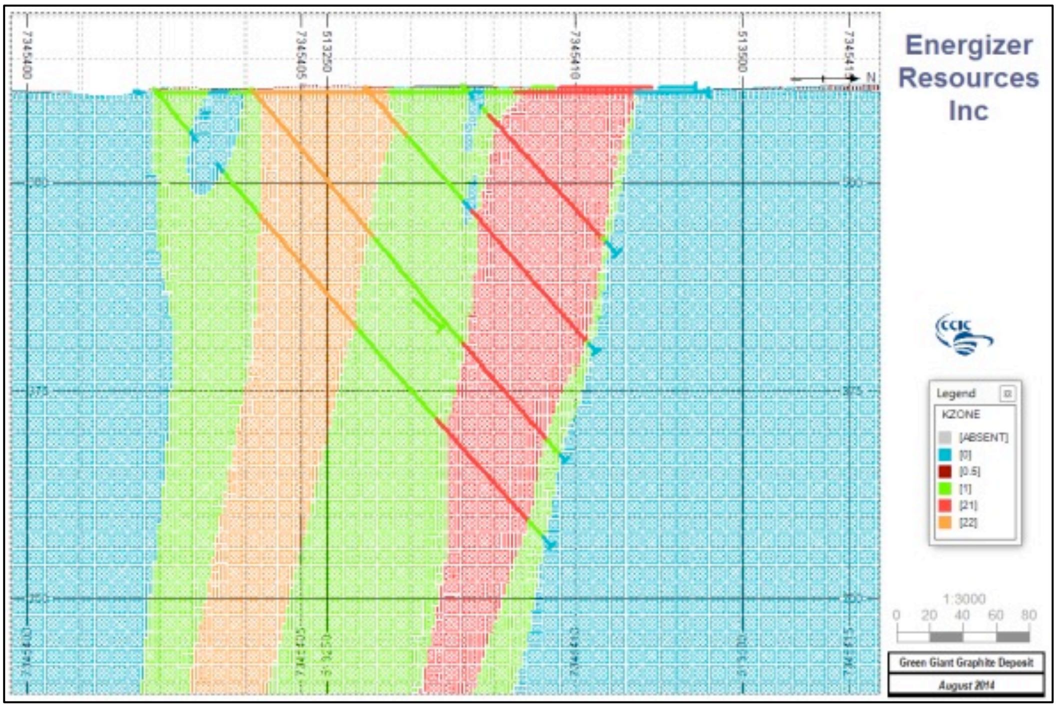


Figure 64: Section Showing Block Model Coding

14.6.4 *Kriging Parameters*

The method for estimating % C was ordinary Kriging. Zonal control was applied, thereby ensuring that samples from a particular domain were constrained to estimating grades only into the block model for that particular domain. The boundaries between the waste and low grade domains were treated as hard boundaries because it is a distinct lithological contact. Boundaries between the low grade and high grade domains were treated as a soft boundary due to their gradational nature.

Parental cell estimation was used. An expanding search ellipse allowed for cells that were not estimated with the minimum estimation parameters, to be estimated. A second search ellipse was expanded by two times. The following fields are recorded in the estimated block model:

C – Ordinary Kriged estimate for % C.

KVAR – Estimated Kriging Variance for % C.

SVOL – Flag to identify which of the two search ellipses was used for the % C estimate.

NUMSAM – Records the number of samples used to estimate % C.

14.7 Model Validation

Model validation included the following:

Visual comparisons of the estimated grades against the composite sample grades.

Statistical comparisons for the mean of estimated grades against the mean of the composited samples.

Trends, (or swath analysis checking) to ensure that the regional grade trends from the drill holes are present in the model. To reduce the estimation errors ordinary Kriging tends to have a smoothing effect on the estimates. The objective of this exercise is therefore to ensure that both regional and local trends are best preserved.

A statistical comparison between the composited samples and the model estimates is presented in Table 31 below. The mean of the samples is weighted by length, while the model estimates are weighted by volume. The means between sample and model estimates compares favourably.

Table 31: Statistical Comparison between Composited Samples and Model Estimates

Sample Composites				**Model Estimates**		
Kzone	LG	HG-E	HG-W	LG	HG-E	HG-W
Field	C	C	C	C	C	C
NSamples	3312	1269	2359	56940	21348	16534
Minimum	0.00	0.08	0.13	0.20	0.74	3.85
Maximum	14.00	15.00	15.00	11.60	11.92	11.69
Mean	**4.68**	**7.75**	**8.36**	**4.58**	**7.56**	**8.17**
Variance	4.41	5.74	6.73	1.71	1.64	1.55
StandDev	2.10	2.40	2.59	1.31	1.28	1.25
Skewness	0.95	-0.00	-0.36	0.39	-0.24	-0.17
Kurtosis	2.20	-0.09	-0.10	1.51	1.40	-0.19
CoV	0.45	0.31	0.31	0.29	0.17	0.15

Block on block analysis compares local trends in the samples against model estimates. The approach here was to divide the study area into 10m*40m*10m blocks in the X, Y and Z direction respectively and to select samples within each block, and compare their mean against the mean of the model estimates within that same block, per KZONE. Plots comparing the mean of the samples (blue) and mean of the estimates (red) for the high grade domains are illustrated in Figure 65, Figure 66 and Figure 67. The mean of the estimates is smoother and less variable than that of the samples, while preserving the grade trends of the samples.

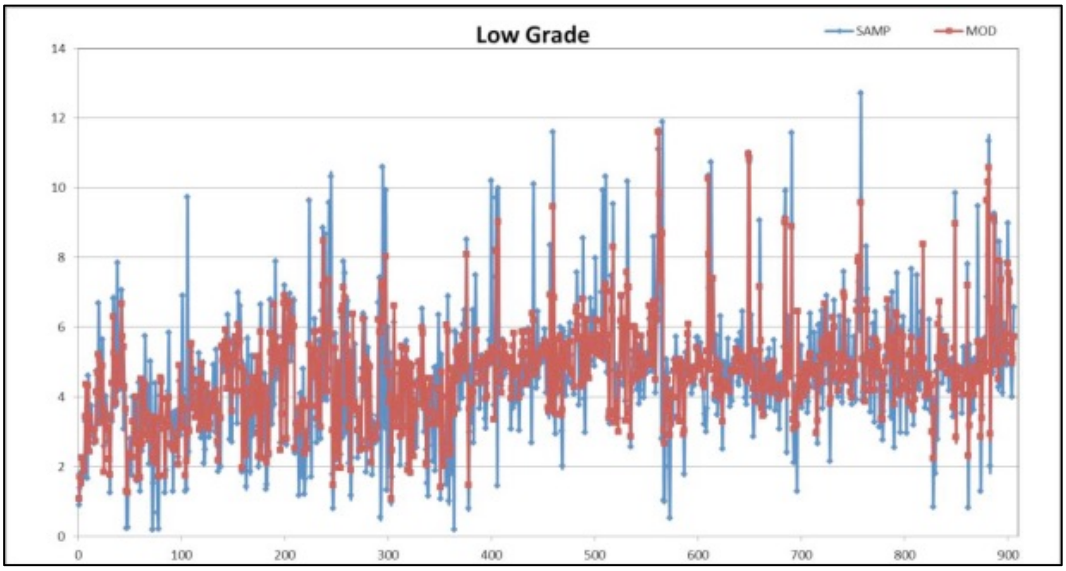


Figure 65:Trend Analysis plot for %C, “low” grade domain

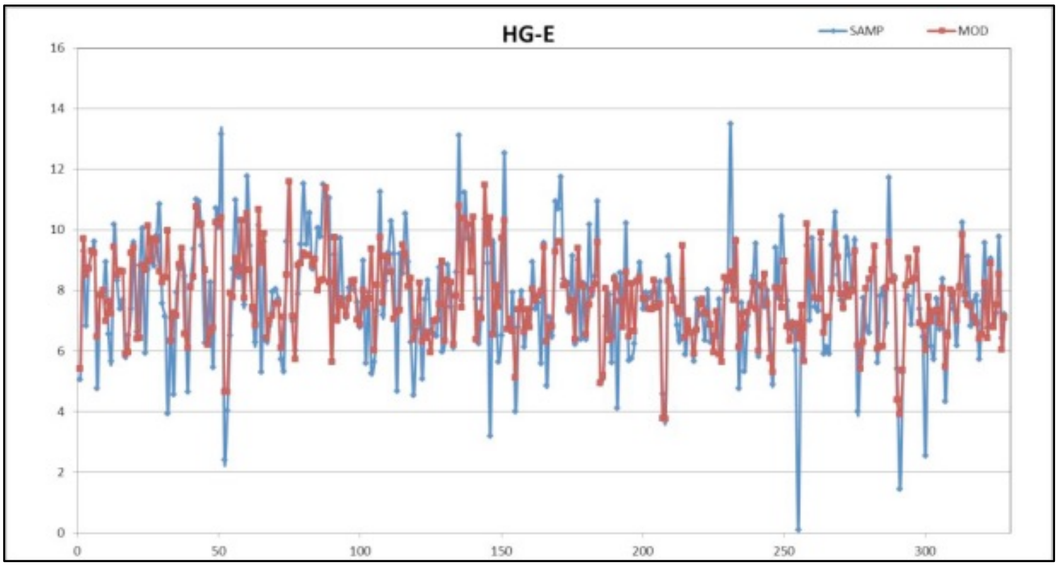


Figure 66:Trend Analysis plot for %C, “High Grade East” domain

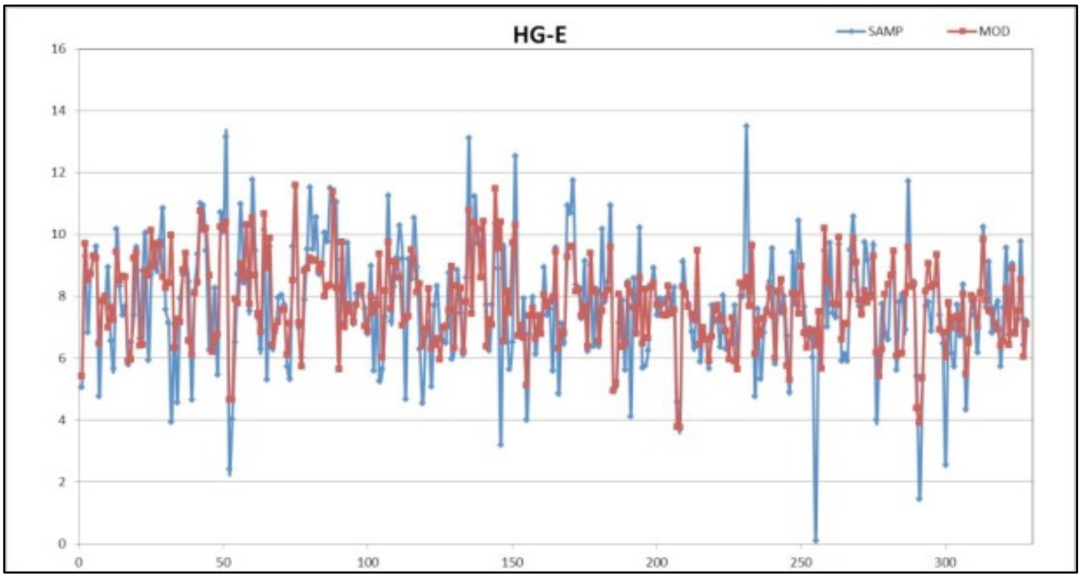


Figure 67: Trend Analysis plot for %C, "High Grade West" domain

14.8 Grade Distribution Plots

Figure 68 is a section illustrating the distribution of the block model super imposed on the drill holes. There is a good correlation between the grades in the drill holes and that of the block model.

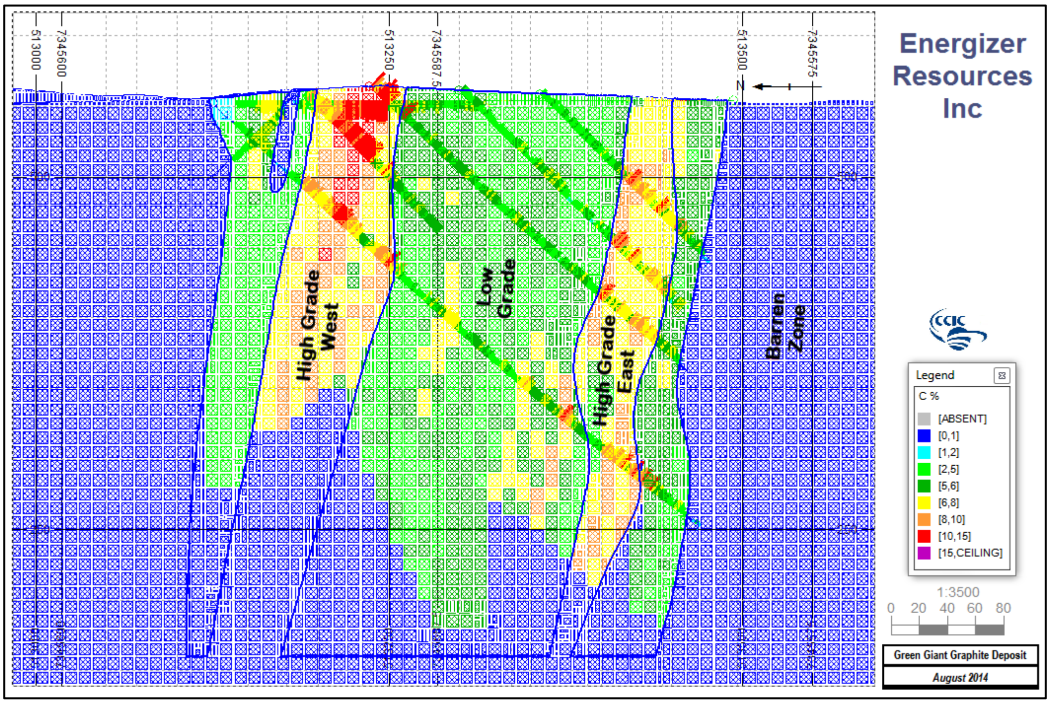


Figure 68: Section Showing C (%) Grade Distribution in the Model

The grade tonnage curves for the "low" and "high" grade domains are presented in Figure 69 and Figure 70. For the "low" grade domain, there is very little variation below a 2% C cut-off, thereafter there is a consistent drop in tonnages and corresponding increase in grade, up to 6.5 % C. For the "high" grade domain, there is very little variation below 4% C, with the majority of this domain occurring between the 6.0 to 10.0 % C range.

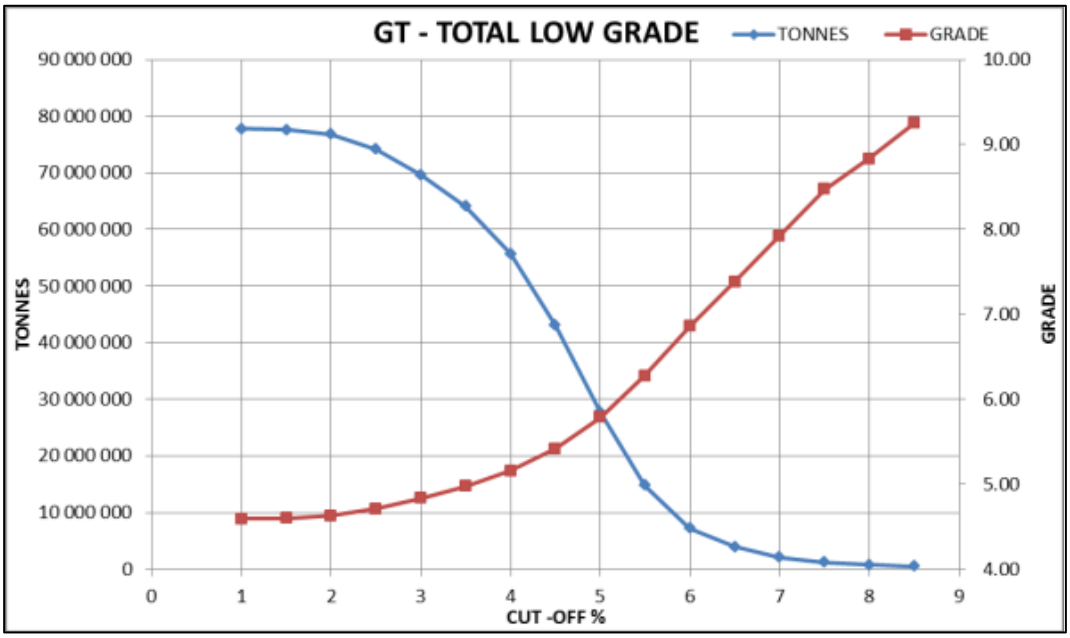


Figure 69: Grade Tonnage Curve for the "Low" grade domain

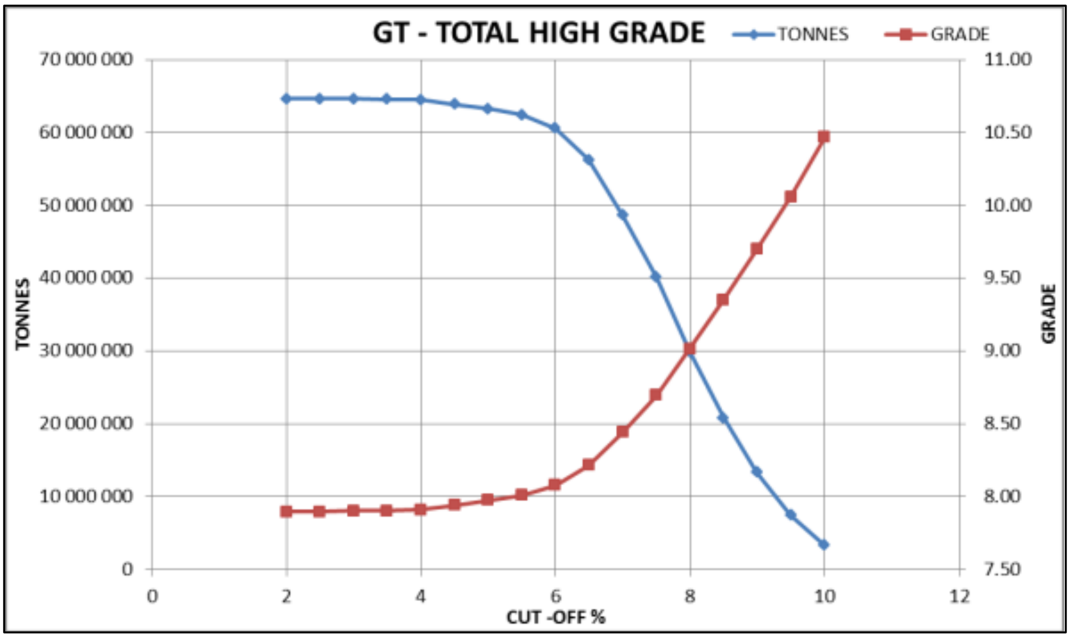


Figure 70: Grade Tonnage Curve for the "High" grade domains

14.9 Resource Classification

The definition of a mineral resource according to the Canadian Institute of Mining ("CIM") reporting code is:

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

Mineral Resources are sub-divided, and must be so reported, in order of increasing confidence in respect of geoscientific evidence, into Inferred, Indicated, or Measured categories.

Based on the geological data and information presented in this report, there is sufficient information about the location, shape, size, geological characteristics and continuity of the deposit to declare a resource.

QA/QC protocols and results indicate an acceptable level of confidence in the analysis of the samples for these drill holes and trenches. There is also a reasonable correlation between drill hole and trench data.

Drill hole spacing varies from 100m by 100m in some areas, to 50m by 50m in other areas especially within the first five years of mining footprint. The well-informed areas provide adequate geological confidence to place the resources into the Measured Category. Hence, the resource classification methodology (Figure 71 and Figure 72) was based on the following criteria:

Areas with drill spacing less than 50m by 50m, was considered for Measured Resources.

Areas with drilling of 50m along dip and 10m along strike, was considered for Indicated Resources.

Areas within the 100m by 100m drill spacing, was considered for Inferred Resources.

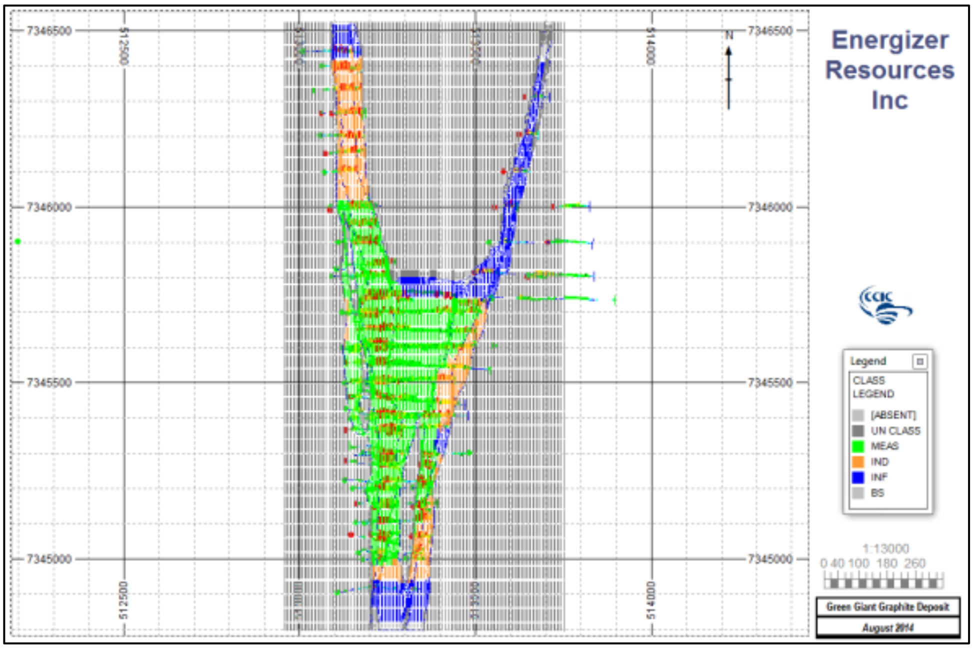


Figure 71: Plan showing Mineral Resource Classification.

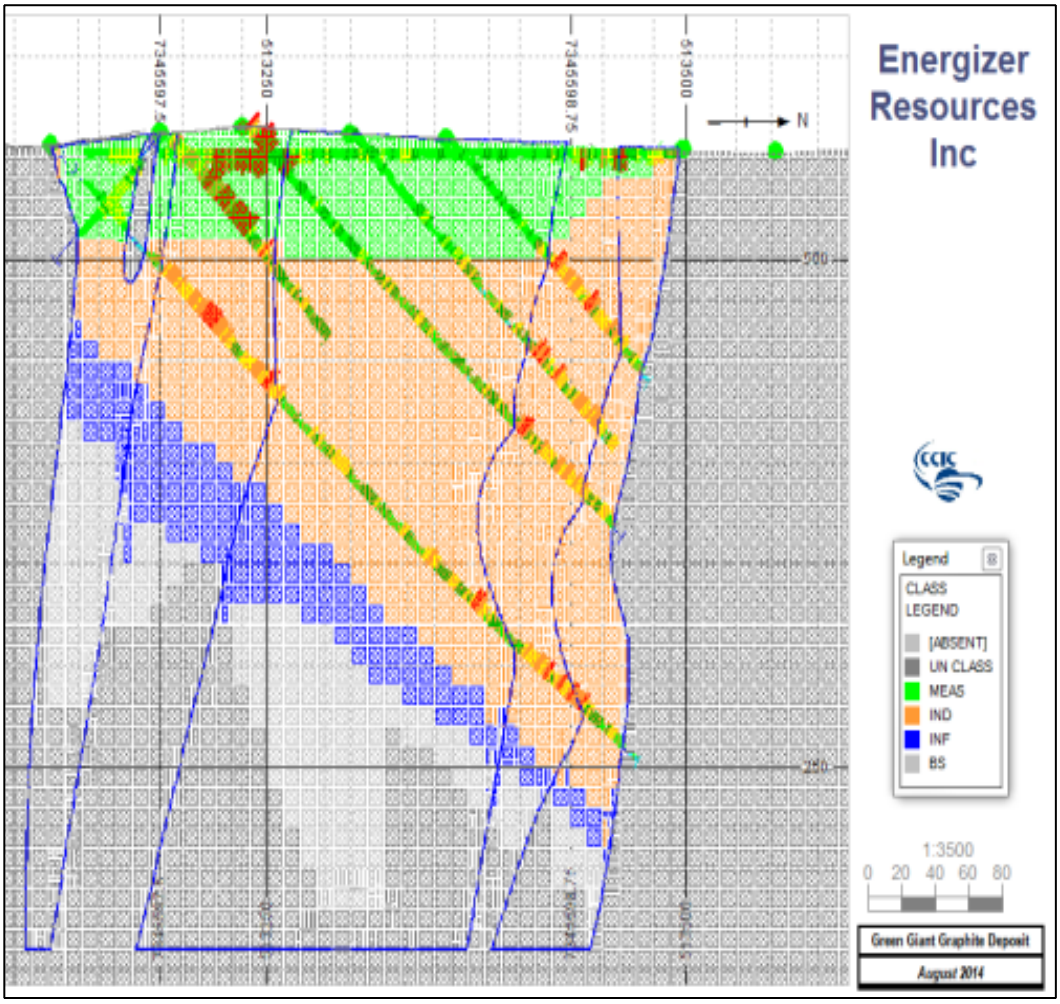


Figure 72: Section showing Mineral Resource Classification

14.10 Resource Tabulation

The current mineral resource estimate for Molo is summarised in Table 32 below. A RD of 2.36 t/m^3 was assigned to the mineralised domains. The mineral resources are classified as Measured, Indicated and Inferred categories according to the CIM Definition Standards. Resources within the low-grade domain are stated at a 2 % C cut-off, and resources within the high-grade domains are stated at a 4% C cut-off. Whilst the “high grade” resources occur within the “low grade” resources, they are estimated and reported separately. The total Measured and Indicated Resources is estimated at 100.37 Mt, grading at 6.27% C. Inferred Resources is at 40.91 Mt, grading at 5.78% C. When compared to the November 2012 resource statement, this shows a 13.7% increase in tonnage 3.4% decrease in grade and 9.8% increase in graphite content. Reason for the increase in tonnage is due to additional exploratory drilling on the north-eastern limb of mineralisation. 23.62 Mt, grading at 6.32% C have been upgraded by infill drilling into the Measure Resource category.

Table 32: Mineral Resource Statement for the Molo Graphite Deposit – September 2014

Classification	Material Type	Tonnes	Grade - C%	Graphite - T
Measured	"Low-Grade"	13 048 373	4.64	605 082
Measured	"High-Grade"	10 573 137	8.4	887 835
Total Measured		23 621 510	6.32	1 492 916
Indicated	"Low-Grade"	39 539 403	4.73	1 871 075
Indicated	"High-Grade"	37 206 550	7.86	2 925 266
Total Indicated		76 745 953	6.25	4 796 341
Measured + Indicated	"Low-Grade"	52 587 776	4.71	2 476 157
Measured + Indicated	"High-Grade"	47 779 687	7.98	3 813 101
Total Measured + Indicated		100 367 464	6.27	6 289 257
Inferred	"Low-Grade"	24 233 267	4.46	1 080 677
Inferred	"High-Grade"	16 681 453	7.70	1 285 039
Total Inferred		40 914 721	5.78	2 365 716

C% = carbon percentage; Graphite – T = Tonnes of graphite

Notes:

Mineral Resources are classified according to the Canadian Institute of Mining definitions.

Mineral Resources are reported Inclusive of Mineral Reserves.

“Low Grade” Resources are stated at a cut-off grade of 2% C.

“High grade” Resources are stated at a cut-off grade of 4% C.

Eastern and western high-grade assays are capped at 15% C.

A relative density of 2.36 tonnes per cubic meter (t/m^3) was assigned to the mineralized zones for the resource tonnage estimation.

15 MINERAL RESERVE ESTIMATES

15.1 Mineral Reserves

The mineral reserves for the Project were estimated by applying a detailed mine design to the Whittle optimisation economic pit shell output which has been drafted on the CCIC prepared resource model and the Whittle input parameters. The CCIC resource model contains Mineral resources that have been reported in the stipulated Measured, Indicated and Inferred categories. Economic, technical factors and a grade cut-off of 4.5% have been applied to the above cut-off Measured and Indicated resources only to report the Molo reserves. The Molo and Molo modular Phase 2 mineral reserves are thus estimated and constrained to the following:

15.2 Whittle Optimisation Shell

15.2.1 Final Pit Design

Using the minimum mining width of 30m suited for the selected mining equipment and the recommended 46° slope angle, the final pit design at Molo was drafted including the key design parameters.

15.2.1.1 Pit Ramps

Three pit ramps namely:

South, central and north ramps have been designed to cater for the full length of the pit and are designed at a width of 15m.

The ramp width is designed to allow dual way traffic and also satisfies the industry norm of the ramp width at least 3.5 times the width of the largest vehicle. The tipper trucks (20t) at a width of 2.5m are more than suited for these ramps. Allowance in the design has also been made for the use of bigger haul trucks if stage 2 expansion require it. The ramps are designed at a 10% gradient and ramp switchbacks, or turns are designed in on a 180 degree turn with an inner radius of 15m.

15.2.1.2 Bench Design

The Bench design includes a bench height of 8m, a berm width of 6m and a bench slope, or batter angle of 75 degrees as recommended from the detailed geotechnical study.

A push back in year 5 designed at limiting the amount of waste mined in the early years and maximising feed grade to optimise the Project's economics.

The final pit design resulted in a pit that is 1.52 km long, 547m wide and 105m deep. (Figure 73)

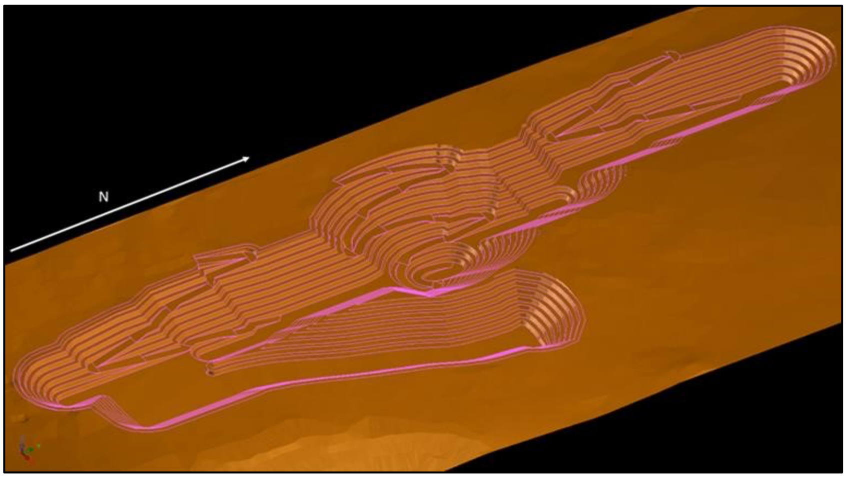


Figure 73: Molo Final Pit Design

15.2.2 *Mining Modifying Factors*

15.2.2.1 *Mining Dilution*

Dilution is waste material that is mined together with ore and thus reduces the overall grade of the mined ore and also increases the tonnages of the RoM.

Despite Molo being a massive deposit regarding its strike, dip, width, and dilution, as a result of mining at the Molo deposit is still expected to be significant due to:

The high-grade zones occur near the outer limits of the deposit and therefore more waste must be mined with depth.

The Molo deposit has an internal waste feature near the central southern section of the deposit and for geotechnical reasons, this waste material must be mined albeit selectively.

The transitional nature of the contact between ore and waste

For typical massive deposit types such as Molo, it is generally accepted that the dilution is 3%. Detailed grade control and dilution test work will be initiated at various bench levels once mining has commenced.

15.2.2.2 *Mining Recovery*

Of the ore that is planned, or scheduled to be mined, a portion of that mineralised material that is in practice left unmined in the pit due to mineralised material / waste boundaries, geotechnical considerations, water in the pit, or other practical issues depending on the nature of the deposit. The mining recovery of the deposits depends on the process methodology and this can range from 82% to 90%. The massive nature of the Molo deposit, the absence of major geotechnical anomalies and the flexibility benefits from using small equipment indicates that the mining recovery can comfortably be set at 95%.

15.2.2.3 *Measured and Indicated Resources*

Mineral reserves for the Molo Project are reported as per NI 43-101 requirements. Only measured and indicated resources can be reported as proven, or probable reserves respectively. For technical and economic reasons, only Measured and Indicated resources that are inside the designed final pit and that are also above the grade cut-off of 4.5% are reported as proven and probable reserves. Inferred resources are not included in the reserve's declaration. All other measured and indicated resources that are below the grade cut-off are also not included in the reserves statement as they are planned to be stockpiled separately as reject material.

15.2.2.4 *Cut-Off Grades*

The cut-off grade is enforced simply to improve the economics of the entire Project, there is thus potential for the rejected material to be treated in the mill later if economic conditions are more favourable. The rejected Measured and Indicated resources inside the pit are 5.1 Mt at an average grade of 3.26% C and these are not included in the reported Proven and Probable reserves.

The Inferred resources inside the final designed pit are 0.96 Mt at an average grade of 5.88% C and these are also not included in the reported reserves.

Table 33: Molo Reserves Statement

Reserve Statement		**As at January 2015**		
Classification	**Material type**	**Ore Tonnes**	**Grade - C%**	**Graphite - Tonnes**
Proven Reserves	High-Grade	9 889 536	7.76%	767 795
	Low-Grade	4 280 205	5.24%	224 374
	Total	14 169 741	7.00%	992 169
Probable Reserves	High-Grade	6 171 268	7.63%	471 069
	Low-Grade	2 095 676	5.31%	111 218
	Total	8 266 944	7.04%	582 287
Grand Total		22 436 685	7.02%	1 574 456
1. All reserves estimated at an economic grade cut-off of 4.5%				

15.3 Factors Affecting the Mineral Reserve Estimate

In the mine design and production scheduling all the factors listed below and described previously have been considered in the final reserve declaration:

Final pit design.

Pit ramps.

Bench design.

Mining dilution.

Mining recovery.

Cut-off grades.

16 MINING METHODS

The surficial, lateral expanse and the massive nature of the Molo deposit make it suitable for open pit mining methods. It is a typical pipe shaped and steeply dipping ore body, with an extended mineral outcrop along the strike (north-south direction) of the deposit. In this mining method, the following activities are executed:

The land is cleared, topsoil is removed and stockpiled at designated sites for use in future land rehabilitation. Depending on the extent of the base of weathering, any further waste, or ore that can be removed by free digging is removed and stockpiled accordingly. The topsoil is planned to be used as a berm around the pit to prevent water flow into the pit and to minimize transportation costs.

In a number of cyclic processes, the waste and/or mineralized material is drilled, charged with explosives and blasted, excavated, hauled and dumped in designated sites.

At strategically planned periods the waste around the boundary of the pit is removed to mine out deeper ore.

The conventional open pit mining activities are carried out with small to medium sized mining equipment including 20t dump trucks, a 2m^3 excavator and an 8m^3 Front End Loader. (Figure 74) below.

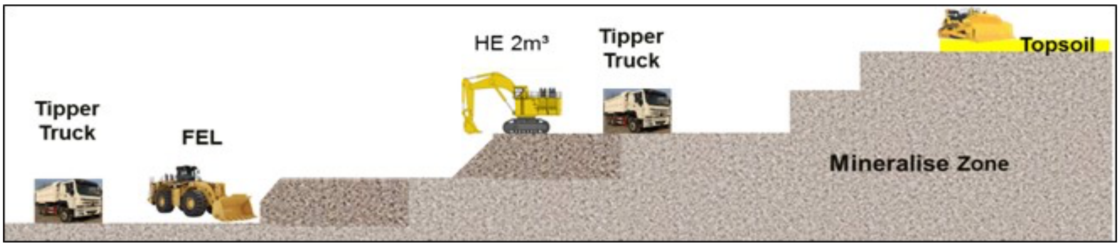


Figure 74: Molo Open Pit Mining Bench Layout

Detailed geotechnical and hydro-geological studies have been conducted and the reports indicate that there are no fatal flaws regarding the adoption of the open pit mining as the preferred mining method.

16.1 Open Pit Optimisation

By using the prepared resource model together with the required economic and technical mine design input parameters, the Whittle Optimization software applies pessimistic and optimistic factors to the product price whilst generating corresponding nested pit shells, or possible pits at their specific total economic values. Whittle executes several possible iterations on the product price, or revenue to obtain the best and optimal pit size and shape in three dimensions.

This optimisation process in the Whittle software is based on the Lerchs-Grossman algorithm. The pit shell that ends up with the highest pit value is then selected as the optimal pit and this pit is established as the basis for any further detailed mine design work.

The Molo resource model has been delineated into two grade zones, high grade and low grade zones. The tonnes and grades of these high grade and low grade resources are reported in the resource statement at 4% and 2% cut-off grades, respectively. Since the graphite product is priced according to the flake size distribution and carbon content, test work studies have been conducted to determine if there are relationships between the grade of the ore fed to the mill and the flake size and carbon content of the graphite product.

These studies initially indicated that the low grade ore yielded larger flake sized product although this was not conclusive and therefore the Whittle optimisation has been run on grade value only. As a result of this the model naturally targets high grade ore early in the LoM. (Table 34) below.

Table 34: Whittle Input Parameters

Mining	Value	Unit
Max bench height	8	M
Minimum bench width	6.14	m
Batter angle	75	deg
Overall Slope angle	46	deg
Minimum pit bottom	30	M
Mining recovery	95	%
Dilution	3	%
Mining operating cost	6.78	US$
Mining re-handle cost	0.56	US$
Processing		
Plant recovery	88.3	%
Mining re-handle cost	0.56	US$
Processing unit cost	19.11	US$
G & A cost	12.64	US$
Mill Throughput	2.35	Mtpa
Product / Selling		
Product price	1 000	US$
Royalty	0%	%
Selling cost	190.55	US$

16.1.1 Pit Slopes

A detailed geotechnical study at the Molo deposit has been completed and the analysis thereof has yielded recommendations for the optimal pit slope angles aimed at maximising extraction and maintaining pit wall stability. Indications are that an overall pit slope angle of 46° could be adopted although the slope angles in the fresh and hard rock material could be made steeper, up to 48°.

16.1.2 Mining Dilution

Dilution is waste material that is mined together with ore and thus reduces the overall grade of the mined ore and increases the tonnages of the RoM.

Despite Molo being a massive deposit regarding its strike, dip, width and dilution due to mining at the Molo deposit is still expected to be significant due to:

The high grade zones occur near the outer limits of the deposit and therefore more waste must be mined with depth.

The Molo deposit has an internal waste feature near the central southern section of the deposit and for geotechnical reasons, this waste material must be mined albeit selectively.

The transitional nature of the contact between ore and waste.

For typical massive deposit types such as Molo, it is generally accepted that the dilution is 3%. Further dilution studies will be investigated at various bench levels once mining has commenced.

16.1.3 Mining Recovery

Of the ore that is planned, or scheduled to be mined, a portion of that mineralised material that is in practice left unmined in the pit due to mineralised material / waste boundaries, geotechnical considerations, water in the pit, or other practical issues depending on the nature of the deposit. The mining recovery of the deposits depends on the process methodology and this can range from 87% to 95%. The massive nature of the Molo deposit, the absence of major geotechnical anomalies and the flexibility benefits from using small equipment indicates that the mining recovery can reasonably be set at 95%.

16.1.4 Plant Recovery and Costs

The plant recovery of 88.3% is based on metallurgical test work results for the Molo samples.

The mining cost is based on owner mining equipment scenario where detailed and up to date quotations by potential suppliers were sourced.

The product price is quoted as US$1,000. The lower plant recovery attempts to maximise the low-grade ore which is expected to produce larger flake sized product.

16.1.5 Open Pit Final Design

Using the minimum mining width of 30m suited for the selected mining equipment and the recommended 46^{o} slope angle, the final pit design at Molo was drafted including the following key design parameters.

16.1.5.1 Ramps

A single ramp was designed in the centre zone of the pit and is designed at a width of 15m. The single ramp is sufficient access to the pit due to the low mining volumes and total mineable volume within the designed pit shell. The ramp width is designed to allow dual way traffic and satisfies the industry norm of the ramp width at least 3.5 times the width of the largest vehicle. The (20t) tipper trucks at a width of 2.5m are more than suited for these ramps. Allowance has also been made for the use of bigger haul trucks if needed.

The ramps are designed at a 10% gradient and ramp switchbacks, or turns are designed in a 180^{o} turn with a radius of 15m.

16.1.5.2 Bench Design

The Bench design includes a bench height of 8m, a berm width of 6m and a bench slope, or batter angle of 75° as recommended from the detailed geotechnical study.

16.2 Mine Planning

16.2.1 Open Pit Optimisation and Conceptual Mine Plan Undertaken for the Anticipated Phase 2 Production Levels

Pit optimisation investigations were carried out on the geological block model "finmod_v4 (1)" to determine whether the Molo operation can sustainably deliver at the Phase 2 production rate of 150 ktpa of carbon concentrate.

16.2.2 The Pit Optimisation Approach

The pit optimisation investigation used appropriate mining software, Studio MPV S, based on the Lerchs-Grossman algorithm. The software determines a series of nested shells using the optimisation parameters presented in Table 35 and an incremental price factor approach. The preferred pit shell is selected based on profitability and its perceived acceptable risk.

Table 35: Pit Optimisation Parameters (Source Sound Mining 2022)

Parameter	Unit	Value
Price		
Graphite Price	US$/t Product	1,230.50
Discount Rate		
Discount Rate	%	10%
Selling Cost		
Royalty	%	1.5%
	US$/t Product	19.05
Transport to Port	US$/t Product	133.01
Selling Cost	US$/t Product	200
Total Selling Cost	US$/t Product	352.06
Processing		
Planned Steady State Production	tpa	2,600,000
Processing Cost	US$/Feed	16.32

Plant Recovery	%	88.3%
Product Grade C	%	97.3%
Mining Cost and Factors		
Mining Variable Cost	US$/mined	4.9
Mining Rehandling Cost	US$/RoM Ore	0.45
Admin Fixed G&A Cost	US$/RoM Ore	4.55
Mining Recovery	%	95%
Mining Dilution	%	3%
Incremental Bench Cost	US$/bench/t	0.007
Pit Geometry		
Slope Angle		46
Bench Height	M	10

Figure 75 and Figure 76 provides isometric views of the geological block model which indicates different mineralised zones characterised as high grade and low grade Mineral Resources. The Mineral Resource statement reported a high-grade cut-off grade of 4% and a low grade cut-off of 2%.

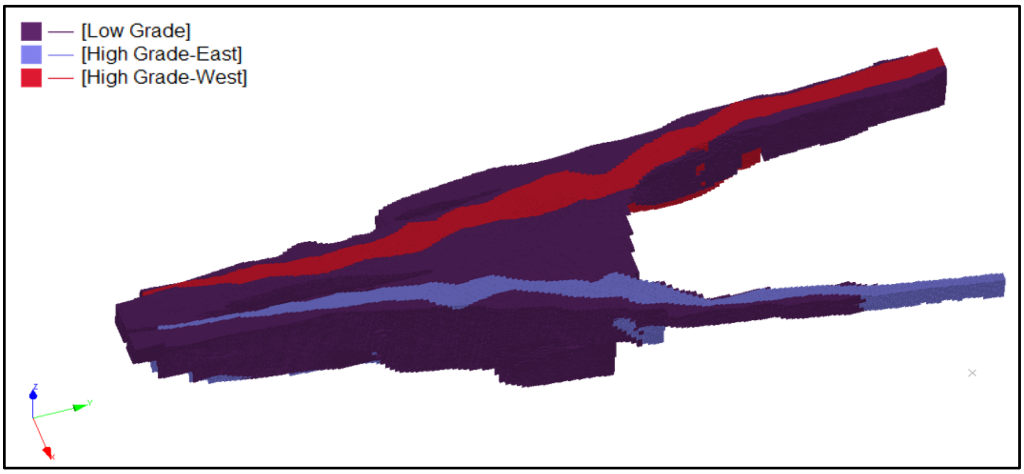


Figure 75: Isometric View of the Block Model (Source: Sound Mining, 2022)

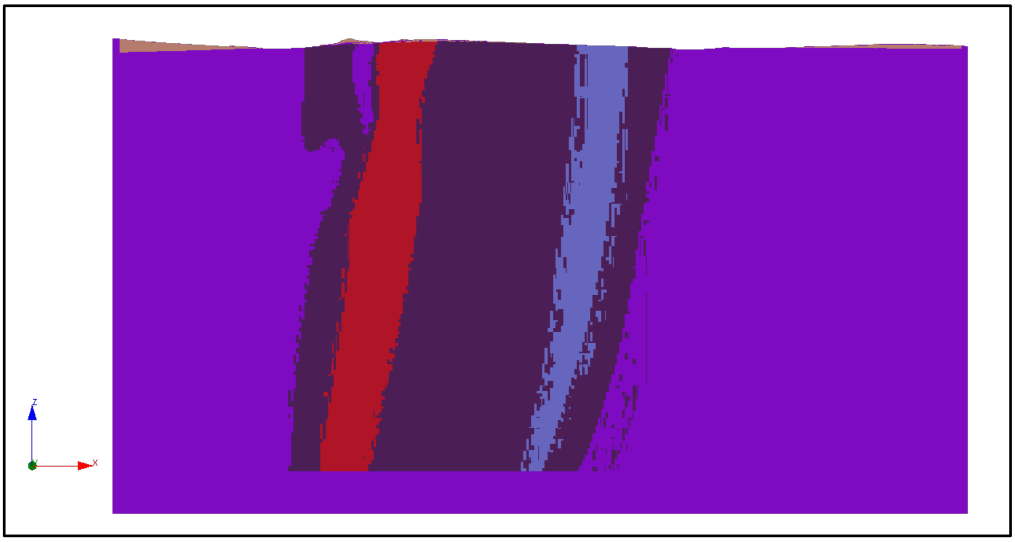


Figure 76: Typical Section through the Block Model showing the Different Mineralised Zones (Source: Sound Mining, 2022)

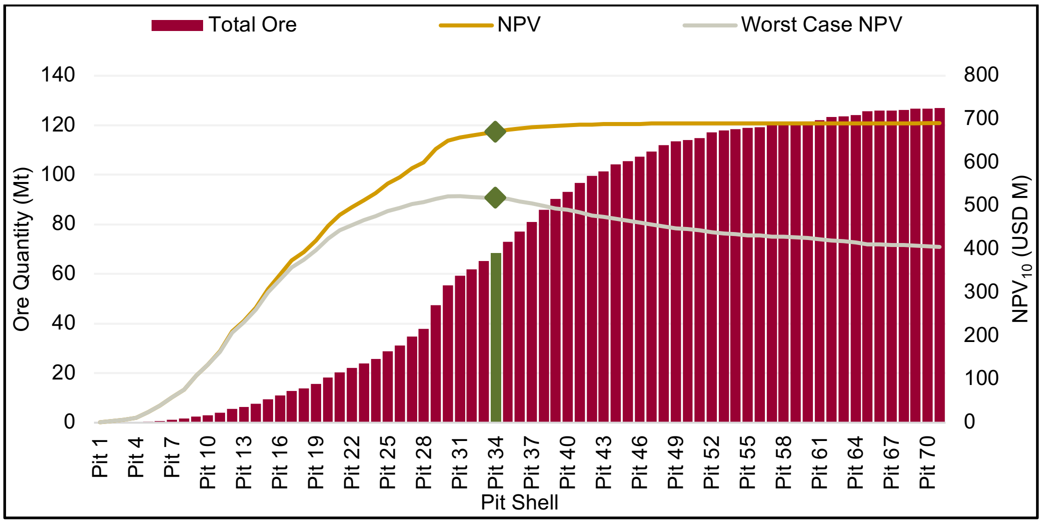
16.3 Pit Optimisation Results

Two pit optimisation scenarios were investigated, namely Scenario 1 and Scenario 2.

16.3.1 Scenario 1

Scenario 1 is unconstrained and as a result all ore materials, including ore from the Measured, Indicated and Inferred Mineral Resource categories, are sent to the treatment plant and no cut-off grade constraints were applied to the plant feed.

Graph 4 presents the derived pit optimisation results. Pit Shell 34 was selected as the preferred pit and considered to represent a practical compromise for the final pit shell selection.



Graph 4: Pit Optimisation Results showing the Selected Pit Shell 34 (Source: Sound Mining, 2022)

Note: Green Diamond Symbol = NPV point for Pit 34

The pit optimisation model indicates that the pit comprises a strike length of approximately 1,700m and a maximum pit width of approximately 500m on the northern limb of the Molo deposit. Figure 77 provides an isometric view of Pit 34 along with its key dimensions.

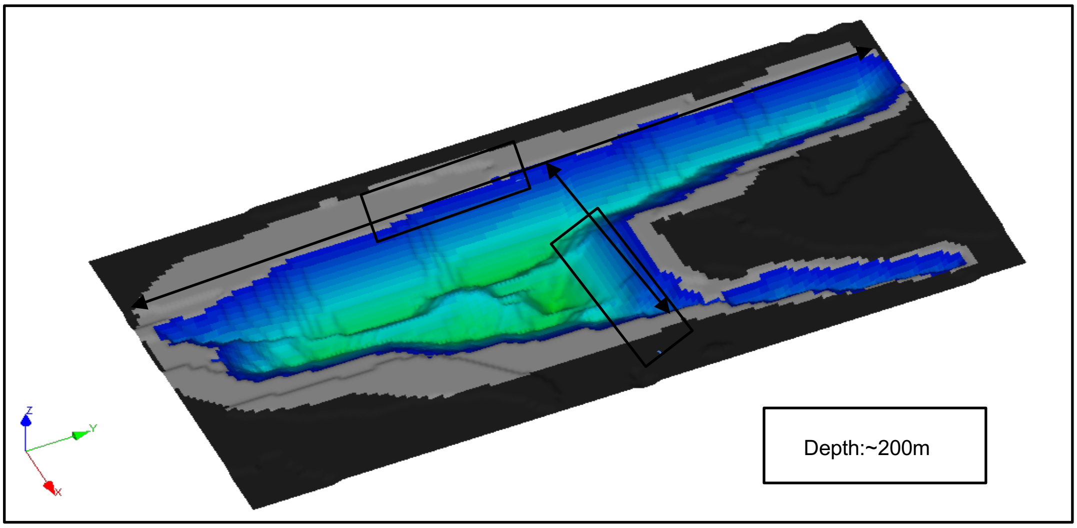


Figure 77: Isometric View of the Selected Pit Shell for Scenario 1 (Source: Sound Mining, 2022)

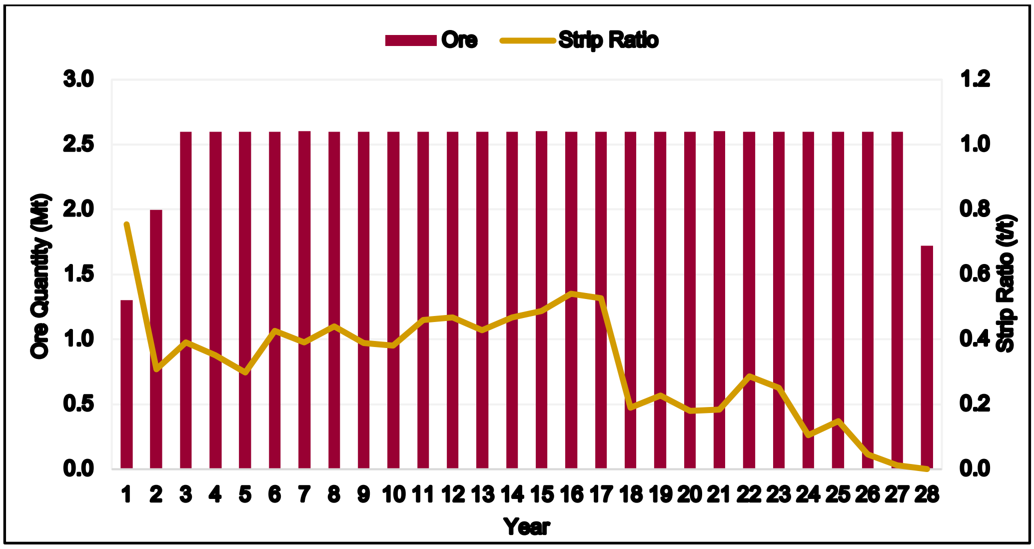
Table 36 provides a summary of the Pit Shell 34 optimisation results, which indicates a total contained pit ore of 68.5Mt at an average grade of 5.60%. This is generated at an average strip ratio of 0.35 (t/t).

Table 36: Scenario 1 Pit Optimisation Results (Source: Sound Mining, 2022)

Description	Unit	Amount
Total Rock	Mt	92.62
C Total in Ore	Mt	3.84
LG Measured	Mt	13.09
LG Indicated	Mt	16.46
LG Inferred	Mt	3.43
HG E Measured	Mt	2.78
HG E Indicated	Mt	9.61
HG E Inferred	Mt	1.91
HG W Measured	Mt	7.79
HG W Indicated	Mt	13.28
HG W Inferred	Mt	1.67

LG Measured-C Grade	%	4.38%
LG E Indicated-C Grade	%	4.53%
LG E Inferred-C Grade	%	4.69%
HG E Measured-C Grade	%	7.43%
HG E Indicated-C Grade	%	7.42%
HG E Inferred-C Grade	%	7.41%
HG W Measured-C Grade	%	8.17%
HG W Indicated-C Grade	%	7.80%
HG W Inferred-C Grade	%	8.02%
Ore	Mt	68.51
Mined Waste	Mt	24.11
Strip Ratio	t/t	0.35

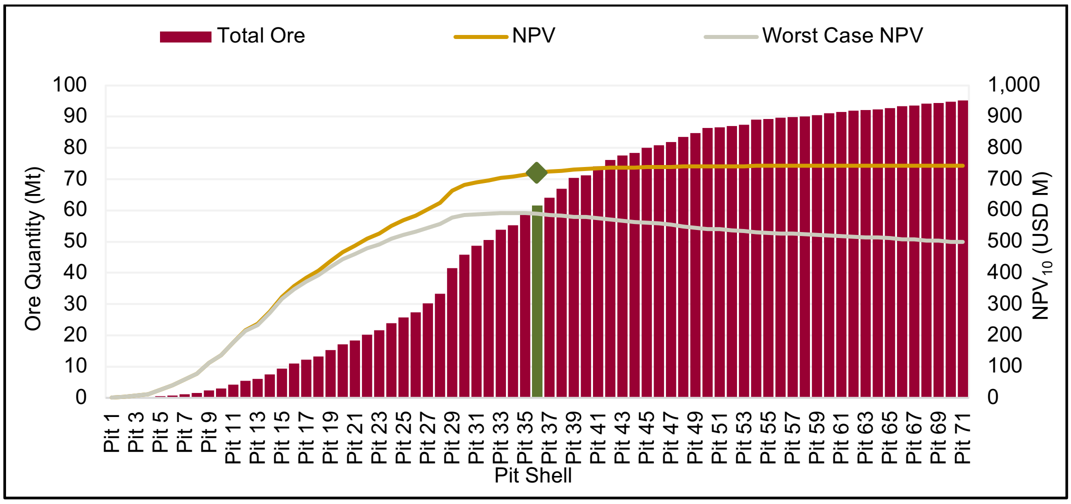
Graph 5 shows a conceptual LoM production schedule over 28 years at a steady state production rate of 2.6 Mtpa.



Graph 5: Conceptual LoM Production Schedule for Scenario 1 (Source: Sound Mining, 2022)

16.3.2 Scenario 2

Scenario 2 is constrained and as a result all ore materials, including ore from the Measured, Indicated and Inferred Mineral Resource categories, are sent to the treatment plant, but a cut-off grade of 4.5% carbon has been applied. All material below this cut-off grade was treated as waste. Graph 6 presents the derived pit optimisation results. Pit Shell 36 was selected and considered to represent a practical compromise for the final pit shell selection.



Graph 6: Conceptual LoM Production Schedule for Scenario 2 (Source: Sound Mining, 2022)

Note: Green Diamond Symbol = NPV point for Pit 36

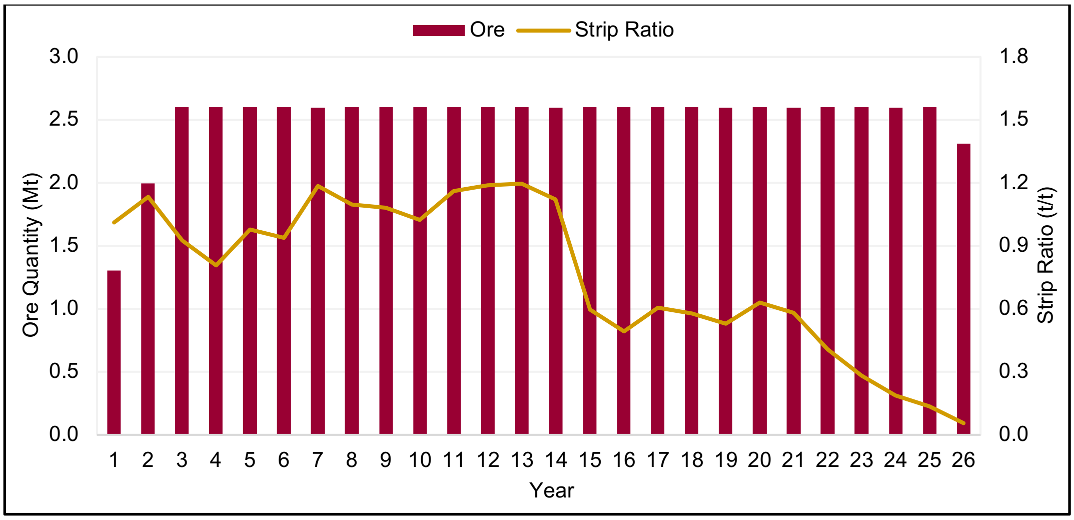
Table 37 provides a summary of the Scenario 2 pit optimisation results. This indicated a total contained pit ore of 61.6 Mt at an average grade of 6.16%. This is generated at an average strip ratio of 0.75 (t/t), which is higher than Projected in Scenario 1. This is a consequence of the cut-off grade inclusion.

Table 37: Scenario 2 Pit Optimisation Results (Source: Sound Mining, 2022)

Description	Unit	Amount
Total Rock	Mt	108.02
C Total in Ore	Mt	3.80
LG Measured	Mt	7.22
LG Indicated	Mt	13.38
LG Inferred	Mt	1.88
HG E Measured	Mt	2.78
HG E Indicated	Mt	11.21
HG E Inferred	Mt	2.05
HG W Measured	Mt	7.79
HG W Indicated	Mt	14.26
HG W Inferred	Mt	2.38

Description	Unit	Amount
LG Measured-C Grade	%	5.13%
LG E Indicated-C Grade	%	5.13%
LG E Inferred-C Grade	%	5.88%
HG E Measured-C Grade	%	7.43%
HG E Indicated-C Grade	%	7.40%
HG E Inferred-C Grade	%	7.35%
HG W Measured-C Grade	%	8.17%
HG W Indicated-C Grade	%	7.78%
HG W Inferred-C Grade	%	7.94%
Ore	Mt	61.61
Mined Waste	Mt	46.41
Strip Ratio	t/t	0.75

Graph 7 shows the conceptual LoM production schedule over 26 years at a steady state production rate of 2.6 Mtpa.



Graph 7: Pit Optimisation Results showing the Selected Pit Shell 36 (Source: Sound Mining, 2022)

16.4 Mining Operations

16.4.1 Mining Production and Support Equipment

To facilitate the mining and material handling required in the mining cycle the following production and support equipment will be required.

Table 38: Mining Production and Support Equipment

Machine / Item	**REPLC. LIFE (hours)**	**Phase 2 Equipment Numbers**
Excavators and Loaders		
Kumatsu PC 1250LC-8B	20,000	1
LIUGONG Front End Loader 3m^3	15,000	2
Haul Trucks		
Kumatsu HD465-7RE	20,000	4
SINOTRUK HOWO Water Truck 20000L (Howa)	15,000	1
Mobile Equipment		
HOWA Truck Mounted Drill Rig	10,000	2
SANY Motor Grader 200HP	15,000	1
SANY Compactor 10t	15,000	1
SHANTUI SD22W HD Tracked Dozer	15,000	2
Service and Support Equipment		
Buses 60-seater	6 years	4
Maintenance / Diesel Truck	6 years	1
Pump Diesel 50m^3/hr	6 years	2
Supervision Vehicle	6 years	6
Light Plant		
Spares (2 Years) -		
Total		**27**

To maintain the targeted mining production, a mining equipment selection exercise was done on the basis of the estimated annual operating hours, cycle times and the effective equipment capacities. This exercise encompassed the primary mining equipment including all the loaders, haul trucks, drill rigs and support equipment. With the mine operating on a one 10 hour shift per 5 day week, 8 effective hours per shift and 260 days per annum. The production mining equipment selected is capable of moving 450,000 tpa during the initial two years and then 2,500 000 tpa thereafter.

One truck mounted drill rig was selected to drill the designed 120 mm blast holes and will be capable of drilling, (5 hours per day 5 day a week) approximately 19,500m per annum at a penetration rate of 15m per hour.

The initial haulage distances for the tipper trucks are expected to be approximately 1.2 km, from the open pit to the Rom tip, or 2.0 km the pit to stockpile areas.

16.4.2 Labour

The number of Expats to be used in the Project has been limited to as few as possible. This is due to the mining being relatively simple, and the productivity levels required being achievable considering the conservative scheduling approach.

This will allow adequate time for operator training and a reasonable production build-up, to be executed. (Table 39) below.

Table 39: Mining Labour Compliment for Phase 2

The Company's Staff List: Mining				**M**	**S**	**U**
Personnel Complement Breakdown	**Responsibility**	**Grade**	**No.**			
Management						
Safety and Training						
Safety Officer	Safety	C4	1		1	
Training Officer	Training	C4	1		1	
Safety and Training			2			
Technical Services						
Tech Manager	Technical Services	D3	0			
Surveyor	Technical Services	C4	1		1	
Geologist	Technical Services	D2	1	1		
Mine Planner	Technical Services	C4				
Technical Services			2			
Mining						
Production Foreman	Mining	C4	2		2	
Miner Blaster	Mining	A2	2		2	

Blasting / Grade Assistants	Mining	A2	6			6
Pumping / Cleaning	Mining	A2	2			2
Excavator Operator	Mining	B3	2			2
FEL Operator	Mining	B3	2			2
Truck Operator	Mining	B3	4			4
Drill Operator	Mining	B3	2			2
Dozer Operator	Mining	B3	2			2
Truck W/E Operator	Mining	B3	1			1
Backhoe Operator	Mining	B3	0			0
Comp / Grader Operator	Mining	B3	2			2
Relief Operator	Mining	B3	3			3
Mining			34			
Engineering						
Engineering GES	Engineering	D1	1	1		
Diesel Mechanic	Engineering	C2	4		4	
Auto Electrician	Engineering	C2	2		2	
Boilermaker	Engineering	C1	1		1	
Diesel Assistant	Engineering	A3	3			3
Boiler Assistant	Engineering	A3	1			1
Engineering			12			
Total Mining Staff			**50**	**2**	**14**	**34**

16.4.3 Drill and Blast Design

The drill and blast design parameters are as follows:

- The single design is applicable to both ore and waste:
    - Waste density = 2.65 / Ore Density = 2.36.
    - UCS range = 15 to 32 MPa.

The drill and blast products assumed are:

- Anfo.
- Emulsion.
- Boosters.
- Electric Detonators.
- Detonating cord.
- Basting wire.
- The drill and blast parameters:

- Hole diameter = 120 mm.
- Bench height = 4m.
- Stemming length = 3.0m.
- Burden = 3.6m.
- Spacing = 4.0 m.
- Sub drill = 0.4 m.

- Explosive density 1.15 g/cm^3
- Powder factor = 0.45.
- Charge mass = 26 kg per hole.

16.4.4 Mine Infrastructure

Provision has been made at the plant site for a mining office which will accommodate 6 people and parking facilities outside the office. The mine terrace adjacent to the plant site has provision for 2 equipment service workshops, tire storage, general storage areas, a vehicle wash bay, brake test ramp and parking for the mine fleet. Additional ablution facilities have been allowed for at the mining infrastructure area.

The change house will be a shared facility which will be utilized by all personnel on the mine, including the plant operators, owners, and security personnel.

Unless otherwise specified, all buildings will be prefabricated steel framed buildings installed on a concrete foundation slab. The proposed buildings to be installed on site include:

- 2 x EMV Workshop, wash bay, tire bay, offices, fuel farm and storage yard.
- Mine administration offices.
- Change House including dressing room, laundry, and boiler room (shared).
- Ablution facilities.

16.4.5 Explosives Storage

The contracted explosives supplier shall use an approved system, (Code of Practice) of delivering raw materials and chemicals which should conform to the legal and safety standards on the mine. The raw materials required to obtain the required quality of explosive will be stored separately and blended once required. A demarcated fenced off storage facility shall be used to store the raw materials and chemicals.

An approved mixing machine designed and supplied by the contracted blasting contractor shall be set up on site to produce the daily ANFO requirements. The raw chemicals shall be stored in appropriated shed / silos set up on the mine. All blasting accessories will be stored in approved explosives and accessories magazines. Only daily requirements shall be withdrawn in accordance with the approved procedure.

The mine shall utilize the magazines to be constructed at the mine. The magazines will be capable of storing boosters, detonator cord, safety fuse, packaged explosives, and accessories in accordance with Madagascar laws and regulations.

The entire explosives infrastructure shall be located at a safe site that will meet the regulatory and standard requirements.

A maximum consumption of 77t of explosives per annum is expected during the first two years and then it should increase to 236 tpa.

17 RECOVERY METHODS

This section describes the process design basis adopted to define the concentrator process design criteria, develop the mass and water balance, and identify and size the major equipment required to process the Molo graphite ore deposit in Madagascar in accordance with the mining design basis set out in Section 16 above.

The ore processing circuit consists of three-stage crushing followed by primary milling and classification, a flotation separation and concentrate upgrading circuit, and final graphite product and tailings effluent handling facilities.

The process is designed, based on metallurgical test work conducted, for an expected overall graphite recovery of 88.3% to final concentrate, at the required grade, from an average plant feed head grade of 7.0%. This will produce an estimated average of 150 ktpa of final concentrate over the LoM, at the design throughput tonnage of 2.5 Mtpa. Negative variations in graphite head grade could be expected to lead to reduced recoveries and/or less graphite product for the same feed tonnage into the plant.

17.1 Process Description

The Molo graphite processing concentrator plant consist of conventional crushing, milling and flotation circuits including the concentrate, dewatering, bagging and tailings processing circuits. The Molo graphite concentrator plant shall consist of the following areas:

- Ore receiving and primary crusher circuit.
- Screening, secondary, and tertiary crusher circuits.
- Primary milling and flash flotation.
- Rougher flotation.
- Primary concentrate cleaning.
- Fine flake cleaning.
- Attrition cleaning circuit.
- Secondary attrition cleaning circuit.
- Final tails handling and disposal.
- Final tails filtration and disposal.
- Final concentrate handling circuit.
- Concentrate drying and bagging.
- Reagents mixing and dosing sections.
- Air and water services.

The RoM material gets crushed in three crushing stages namely, primary, secondary, and tertiary crushing. The material then goes through primary milling, flash, and rougher flotation. The material then gets subjected to primary concentrate cleaning, fine flake cleaning and attrition cleaning (alternatively secondary attrition cleaning).

The concentrate material gets thickened to produce a concentrate filter cake, which will be dried to reduce the moisture content to less than 20%. The concentrate gets screened into four different size fractions and bagged separately as final product.

The tailings material gets thickened and filtered to produce a tailings filter cake, which is then stored away.

17.2 Ore Receiving and Crushing

The RoM is fed into the ore receiving and primary crushing circuit via a Front-end Loader ("FEL"). The RoM material is fed onto the RoM static grizzly screen 1100-SR-004. The RoM static grizzly is fitted with a dust suppression system. The RoM static grizzly is used as a scalping screen to control the size of material to the primary jaw crusher 1110-CR-100. The material gets screened using a static grizzly screen situated at the top of the primary crusher bin 1100-BN-006 to reject rocks larger than 300 mm as the oversize material. The oversize material gets stockpiled near the feed bin and may be manually crushed using rock breakers.

The static grizzly undersize material less than 300 mm discharges into the primary crusher bin 1100-BN-006. The primary crusher bin discharges the material on the grizzly feeder 1100-FD-008. The material gets further screened using the grizzly feeder fitted with 70 mm openings. The grizzly feeder is fitted with a dust suppression system.

The undersize material from the grizzly feeder less than 70 mm is discharged on the primary jaw crusher discharge conveyor belt 1100-CV-011. The oversize material from the grizzly feeder greater than 70 mm is discharged into the primary jaw crusher 1100-CR-010. The crusher is set to a CSS value of 75 mm and is expected to produce 100% -120 mm material. The crusher product and the grizzly undersize are discharged onto the primary crusher discharge conveyor 1100-CV-011. The primary crusher discharge conveyor belt is fitted with a dust suppression system.

The primary crusher product conveyor belt discharges on to the classification screen feed conveyor belt 1100-CV-012. The classification screen feed conveyor belt is fitted with a dust suppression system, a tramp metal magnet 1100-EE-046 as well as a weightometer. The classification screen feed conveyor feeds into to the double deck classification screen 1100-SR-040. The classification screen is fitted with two decks; the top deck is fitted with 35 mm screen panels and the bottom deck is fitted with 16 mm panels. The oversize from the top deck greater 35 mm discharges onto the secondary crusher feed bin 1100-BN-022. The material from the secondary crusher feed bin is drawn via the secondary crusher feeder 1100-FD-024 on to the secondary crusher feed conveyor belt 1100-CV-034. The secondary crusher feed conveyor belt discharges into the secondary crusher 1100-CR-026. The secondary crusher discharges its product onto the classification screen feed conveyor belt.

The undersize from the bottom deck less than 35 mm and greater than 16 mm discharges onto the tertiary crusher feed bin 1100-BN-028. The material from the tertiary crusher feed bin is drawn via the tertiary crusher feeder 1100-FD-030 on to the tertiary crusher feed conveyor belt 1100-CV-036. The tertiary crusher feed conveyor belt discharges into the secondary crusher 1100-CR-032.

The tertiary crusher discharges its product onto the classification screen feed conveyor belt. The products from the secondary and tertiary crushers are combined and recycled back to the classification screen. The bottom deck screen undersize less than 16 mm size fraction from the classification screen gets discharged onto the primary mill feed conveyor 1100-CV-044. The primary mill feed conveyor belt is fitted with a dust suppression system. The primary mill feed conveyor discharges into the mill feed bin 1200-BN-058.

17.3 Primary Milling and Flash flotation

The primary mill feed bin material is drawn via a primary mill feed bin feeder 1200-FD-060. The primary mill feed bin feeder discharges onto the primary mil feed conveyor 1200-CV-066. The primary mil feed conveyor is fitted with hammer sampler 1200-SA-098 and a weightometer 1200-ZM-068.

The primary mill feed conveyor feeds onto the primary mill scalping screen 1200-SR-070. The scalping screen consists of two screening decks; top deck consists of 6 mm aperture panels whilst the bottom deck is fitted with 2 mm slots. Any spillage material in the primary milling circuit is pumped to the scalping screen through the mill area spillage pump 1200-PP-094. The material greater than 2 mm (top and bottom deck oversize), feeds into the primary ball mill 1200-ML-078 by gravity. The primary mill is expected to produce a product with a grind of 80% passing 500 microns (µm). The primary mill product is discharged into the primary ball mill discharge sump 1200-TK-082. The primary mill scats report into the primary mill scatts bin 1200-BN-080.

The material less than 2 mm from the scalping screen flows by gravity into the flash flotation cell. The flash flotation cell is installed to take advantage of graphite liberation that takes place at coarse particle sizes. Flash flotation produces a concentrate that reports to the total concentrate sump 1300-TK-134.The flash flotation tailings discharge into the flash floatation tailings sump 1200-TK-088 and gets pumped to the primary mill discharge sump via floatation tailings sump pumps 1200-PP-090/092.

The slurry from the primary mill discharge sump gets pumped via the primary mill discharge pump 1200-PP-084/086 to the primary mill trash screen 1200-SR-074. The primary mill trash screen oversize reports to the primary mill trash screen bin 1200-BN-084. The primary mill trash screen undersize feeds onto the primary mill linear screen 1200-SR-072, which is fitted with a 600 µm screen cloth. The oversize material of the linear screen greater 600 µm discharges into the primary mill. The undersize material of the linear screen less 600 µm reports to the rougher flotation circuit.

17.4 Rougher Flotation

The rougher flotation surge tank 1300-TK-728 is fed by the primary mill linear screen undersize material feed. The slurry consists of the flash flotation tailings and the mill product. The rougher flotation bank 1300-FC-110/112/114/116/118 by gravity via the rougher flotation surge tank. The frother and collector reagents are added into rougher cell no.1 1300-FC-110 and rougher cell no.3 1300-FC-114. The rougher concentrate discharges into the total concentrate sump 1300-TK-134.The concentrate from flash flotation, primary scavenger concentrate and the primary rougher concentrate are combined in the total concentrate sump and pumped to the primary concentrate cleaning circuit using the pumps 1300-PP-136/137.

The tailings from the rougher flotation circuit are discharged into the combined tailings sump 1300-TK-138. The combined tailings sump feeds from the attrition cleaner tailings, fine flake cleaner, the primary cleaner scavenger tailings, and rougher tailings. The material from combined tailings sump is then pumped to the final tailings thickener 1400-TH-310, via the combined tailings pump 1300-PP-140/142.The spillage from the rougher flotation circuit is pumped to the primary ball mill scalping screen using the rougher flotation spillage pump 1300-PP-144.

17.5 Primary Concentrate Cleaning

The primary cleaner concentrate dewatering screen 1300-SR-146 is fed from the combined concentrate. The primary cleaner concentrate dewatering screen is fitted with 74 µm panels. The oversize material from the primary cleaner concentrate dewatering screen is discharged into the primary polishing mill 1300-ML-148. The primary polishing mill product discharges into the primary polishing mill discharge sump 1300-TK-150.The dewatering screen undersize discharges into the primary polishing mill discharge sump.

The combined material from the primary polishing mill discharge sump gets pumped to the primary cleaner column cell 1300-FC-154 via the primary polishing mill discharge sump pump 1300-PP-152. The primary cleaner column cell concentrate gets discharged onto the primary cleaner column flotation cell concentrate classification screen 1300-SR-190 fitted with 212 µm screen panels. The oversize material from the primary cleaner column concentrate classification screen flows by gravity into the fine flake cleaning circuit. The undersize material of the primary cleaner column concentrate screen discharges into the concentrate classification screen undersize sump 1300-TK-192. The material from the classification screen undersize sump gets pumped to the fine flake cleaning circuit using the primary column cell concentrate undersize pump 1300-PP-196.

The primary cleaner column cell tailings get discharged into the primary cleaner column cell tailings sump 1300-TK-160. The material then gets pumped to the primary cleaner flotation bank 1300-FC-164/166/168 via 1300-PP-162. The concentrate from the primary cleaner flotation bank gets recycled back to the primary polishing mill discharge sump. The tailings from the primary cleaner flotation bank gets discharged into the primary scavenger cleaner flotation bank1300-FC-170/172/174. The concentrate from the primary scavenger cleaner flotation bank gets pumped to the total concentrate sump via the primary scavenger cleaner concentrate sump pump 1300-PP-182. The tailings from the primary scavenger cleaner flotation bank gets discharged into the combined tailings sump.

The final cleaner tailings sump 1300-TK-184 gets fed from the attrition cleaner tailings and the fine flake cleaner tailings. The combined tailings from final cleaner tailings sump gets pumped to the combined tailings sump via the final cleaner tailings sump pump 1300-PP-186.

17.6 Fine Flake Cleaning

The fine flake dewatering screen 1300-SR-267 is fed from the primary column concentrate screen undersize. The fine flake dewatering screen is fitted with 75 µm panels. The oversize material from the fine flake dewatering screen is discharged into the fine flake polishing mill 1300-ML-268. The fine flake polishing mill product discharges into the fine flake polishing mill discharge sump 1300-TK-270. The dewatering screen undersize discharges into the fine flake polishing mill discharge sump.

The combined material from the fine flake polishing mill discharge sump gets pumped to the fine flake cleaner column cell 1300-FC-274 via the fine flake polishing mill discharge sump pump 1300-PP-272. The fine flake cleaner column cell concentrate gets discharged into the combined cleaner concentrate sump 1300-TK-278. The feed to the combined cleaner concentrate sump includes the primary column concentrate screen oversize and the fine flake cleaner column cell concentrate. The combine concentrate from the combined cleaner concentrate sump gets discharged in the attrition cleaning circuit via combined cleaner concentrate sump pump 1300-PP-280.

The fine flake cleaner column cell tailings get discharged into the fine flake cleaner column cell tailings sump 1300-TK-282. The material then gets pumped to the fine flake cleaner flotation bank 1300-FC-286/288/290via 1300-PP-284. The concentrate from the fine flake cleaner flotation bank gets recycled back to the fine flake polishing mill discharge sump. The tailings from the fine flake cleaner flotation bank gets discharged into the fine flake scavenger cleaner flotation bank 1300-FC-292/294. The concentrate from the fine flake scavenger cleaner flotation bank gets discharged into the fine flake scavenger cleaner discharge sump 1300-TK-298. The material then gets pumped back to the fine flake dewatering screen via the fine flake scavenger cleaner discharge sump 1300-PP-300.

17.7 Attrition Cleaning

The attrition concentrate thickener 1300-TH-001 gets fed from the combined cleaner concentrate via the attrition concentrate thickener basket 1300-SR-004. The overflow of the attrition concentrate thickener get discharged into the process water tank. The underflow of the attrition concentrate thickener gets drawn via the attrition concentrate thickener pumps 1300-PP-002/003 to the attrition mill 1300-ML-005.

The product from the attrition mill flows into the attrition mill discharge sump 1300-TK-010.The material from the attrition mill discharge sump gets pumped to the attrition cleaner column cell 1300-FC-020 via the attrition mill discharge pump 1300-PP-015.The concentrate from the attrition cleaner column cell gets discharged into the attrition cleaner column cell concentrate discharge sump 1300-TK-065. The material from the attrition cleaner column cell concentrate discharge sump gets pumped to the final concentrate thickener circuit via the attrition cleaner column cell concentrate discharge sump pump 1300-PP-070.

The tailings from the attrition column cell are discharged into the attrition column cell tailings sump 1300-TK-030. The material from attrition column cell tailings sump gets pumped to the attrition cleaner flotation bank 1300-FC-040/045 via the attrition column cell tailings sump pump1300-PP-035. The concentrate from the attrition cleaner flotation bank gets re-circulated back to the attrition mill discharge sump via gravity.

The tailings from the attrition cleaner flotation bank gets discharged to the attrition scavenger flotation cell 1300-FC-050. The concentrate of the attrition scavenger cleaner flotation cell is pumped to the attrition concentrate thickener via the attrition scavenger cleaner flotation cell pump 1300-PP-060. The tailings from the attrition scavenger cleaner cell are recycled by gravity back to the final cleaner tailings sump in the primary concentrate cleaning circuit. The spillage from the attrition cleaning circuit is pumped to the attrition cleaning thickener via the attrition cleaning spillage pump 1400-PP-204.

17.8 Secondary Attrition Cleaning

The secondary attrition cleaning circuit is an alternative, or bypass circuit to the attrition cleaning circuit.

The secondary attrition classification screen 1300-SR-700 gets fed from the combined cleaner concentrate. The undersize material of the secondary attrition screen gets pumped to the attrition concentrate thickener via the secondary attrition classification screen undersize pump 1300-PP-728. The oversize material from the secondary attrition classification screen is discharged into the secondary attrition classification screen sump. The material from the secondary attrition classification screen sump gets pumped to the secondary attrition mill 1300-ML-702, or it can be bypassed to the attrition mill via the secondary attrition classification screen sump pump 1300-PP-703.

The product from the secondary attrition mill flows into the secondary attrition mill discharge sump 1300-TK-704. The material from the secondary attrition mill discharge sump gets pumped to the secondary attrition cleaner column cell 1300-FC-708 via the attrition mill discharge sump pump 1300-PP-706.The concentrate from the secondary attrition cleaner column cell gets discharged into the secondary attrition cleaner column cell concentrate discharge sump 1300-TK-724. The material from the attrition cleaner column cell concentrate discharge sump gets pumped to the final concentrate thickener circuit via the secondary attrition cleaner column cell concentrate discharge sump pump 1300-PP-726.

The tailings from the secondary attrition column cell are discharged into the secondary attrition column cell tailings sump 1300-TK-710. The material from the secondary attrition column cell tailings sump gets pumped to the secondary attrition cleaner flotation bank 1300-FC-714/716 via the secondary attrition column cell tailings sump pump 1300-PP-712. The concentrate from the secondary attrition cleaner flotation bank gets re-circulated back to the secondary attrition mill discharge sump via gravity.

The tailings from the secondary attrition cleaner flotation bank gets discharged to the secondary attrition scavenger flotation cell 1300-FC-718. The concentrate of the secondary attrition scavenger cleaner flotation cell is pumped to the secondary attrition concentrate thickener via the secondary attrition scavenger cleaner flotation cell pump 1300-PP-722. The tailings from the secondary attrition scavenger cleaner cell are recycled by gravity back to the final cleaner tailings sump in the primary concentrate cleaning circuit.

17.9 Final Tailings Handling

17.9.1 Tailings Handling and Disposal

The final tailings thickener 1400-TH-310 gets fed from the combined tailings sump, via the final tailings thickener basket 1400-SR-311. The overflow of the final tailings thickener reports to the process water tank. The underflow of the final tailings thickener gets pumped to the tailings filter feed tank via the final tailings thickener underflow pumps 1400-PP-312/314. The spillage from the final tailings handling and disposal circuit is pumped to the final tailings thickener, via the final tailings spillage pump 1400-PP-338.

17.9.2 Tailings Filtration and Disposal

The final tailings filtration and disposal circuit is operated in a semi-batch manner and it is anticipated to produce a tailings filter cake with 15% moisture. The final tailings filtration feed tank 1400-TK-040 is fed from the final tailings thickener underflow. The final tailings filtration feed tank is equipped with an agitator 1400-AG-042.

The final tailings thickener underflow material from final filtration feed tank is pumped to the final tailings filter 1400-FL-052 via the final filtration feed tank pumps 1400-PP-048/050. The filtrate from the final tailings filter is discharged into the final tailings filtrate sump 1400-TK-056 and recycled to the final tailings thickener. The filter cake is discharged onto the final tailings filter cake conveyor belt 1400-CV-064. The filter cake is discharged into the final tailings filter cake bunker via the final tailings filter cake conveyor belt 1400-CV-068. The final tailings filter cake is therefore stockpiled and discarded by a frontend loader (FEL). The spillage from the final tailings filtration and disposal circuit is pumped to the final tailings thickener feed tank via the final tailings filtrate spillage pump 1400-PP-070.

17.10 Final Concentrate Handling

17.10.1 Concentrate Handling

The final concentrate thickener 1500-TH-342 is fed from the combined concentrate sump via the final concentrate thickener basket 1500-SR-343. The final concentrate thickener overflow is discharged into the process water tank. The final concentrate thickener underflow is pumped to the final concentrate belt filter feed tank 1500-TK-350 via the final concentrate thickener underflow pumps 1500-PP-344/346. The final concentrate belt filter feed tank is equipped with an agitator 500-AG-348.

The final concentrate thickener underflow is pumped to the final concentrate belt filter 1500-FL-358 via the final concentrate belt filter feed tank pump 1500-PP-352. The final concentrate belt filter is equipped with a final concentrate filter vacuum pump 1500-PP-360. The final thickener underflow is filtered in the final concentrate belt filter that is expected to produce a filter cake with less than 20% moisture. The filtrate from the final concentrate belt filter is pumped back to the final concentrate thickener via the final concentrate belt filter filtrate pump 1500-PP-370.The filter cake from the final concentrate belt filter is discharged onto the final concentrate filter cake conveyor 1500-CV-374 and fed to the final concentrate drying and bagging circuit.

The spillage from the final concentrate handling circuit is pumped to the final concentrate thickener via the final concentrate filtrate spillage pump 1400-PP-340.

17.10.2 Concentrate Drying and Bagging

The concentrate from the final concentrate filter cake conveyor is fed into a diesel-fired dryer 1500-DR-376.The diesel-fired dryer produces the fine dry graphite concentrate powder and the coarse dry graphite concentrate flakes. The dry concentrate feeds into the concentrate surge bin 1500-BN-380. The dry concentrate from the concentrate surge bin is drawn via the fine concentrate pneumatic pump 1500-PP-348 to the dried concentrate bin. The bucket elevator 1500-FD-392 is fed from the dried concentrate bin rotary valve 1500-FD-388. The dried concentrate is screened into four size fractions namely:

+400 µm, -400+177 µm, -177 µm +149 µm and -149 µm. Each of the size fractions are bagged separately as final product. The bucket elevator feeds into the double deck sifter screen with the top deck screen panels of 400 µm and bottom deck of 177 µm.

The oversize material of the top deck reports to the 400 µm bagging plant and the oversize material of the bottom deck re-pots to the 177 µm bagging plant. The undersize of the double deck sifter screen feeds onto the single deck sifter screen equipped with 149 µm screen panels.

The oversize material of the single deck sifter screen reports to the 149 µm bagging plant. The undersize material of the single deck sifter screen reports to the 149 µm bagging plant.

17.11 Water Services

The process water tank 1600-TK-472 is fed from the final concentrate thickener overflow, attrition concentrate thickener overflow, and the tailings thickener overflow.

Make-up raw water is pumped from a well-field into the well field collection tank 1600-TK-450. The raw water from the well field collection tank feeds into the process water tank, plant raw water tank and to the potable water plant 1600-ZM-454 via the well field collection tank pumps 1600-PP-466/464 respectively.

The storm / pollution control dam 600-TK-478 feeds into process water tank

The water is distributed to the following plant areas from process water tank using process water pumps1600-PP-474/476:

- Attrition cleaning.
- Fine flake cleaning.
- Primary milling and flotation.
- Rougher floatation.
- Primary concentrate cleaning.
- Secondary attrition cleaning.

Treated water from the potable water plant is stored into the potable water storage tank 1600-TK-456. The water from the potable water storage tank is drawn via the potable water storage tank pumps 1600-PP-458/460 to the potable water camp and to the potable water plant respectively.

The plant potable water tank is fed via the potable water storage tank pump 1600-PP-460. The potable water plant is fed via the plant potable water tank pump 1600-PP-466.

The raw water tank is fed via the plant potable water tank pump 1600-PP-466. The plant raw water tank distributes raw water for dust suppression, gland service water and fire water purposes via the raw water tank pumps to the following areas:

Raw water tank pump 1600-PP-432:

- Flocculant plant.
- Ore receiving dust suppression.
- Secondary / tertiary crusher dust suppression.

- Coagulant plant.

Raw water tank pump 1600-PP-436/438:

- Primary milling gland service water.
- Rougher flotation gland service water.
- Primary cleaner gland service water.
- Tailings thickener underflow pumps gland service water.

Raw water tank pump 1600-PP-448/552:

- Fire water.

17.12 Air Services

The flotation blower 1600-HB-484 distributes air to the following circuits:

- Flash flotation.
- Rougher flotation.
- Primary cleaning.
- Fine flake cleaning.
- Attrition cleaning.
- Secondary attrition cleaning.

The instrument air compressor 1600-HA-488 distributes air to the instrument dryer 1600-DR-492 and the column air receiver 600-TK-498. The instrument dryer feeds into the instrument air receiver 1600-TK-496 which provides instrument air. The column air receiver distributes air to the following equipment:

- Primary column cleaner.
- Fine flake column.
- Attrition cleaning column.
- Secondary cleaning column.

The tailings filter air compressor 1600-HA-491 feeds into the tailings filter air receiver 1600-TK-499 which provides tailings filtration air.

17.12.1 Collector Storage and Dosing

The collector storage isotainer 1800-TK-500 distributes the frother to the following equipment via the following pumps:

- Flash flotation collector via the collector pump 1800-PP-502.
- Rougher flotation cell #1 collector via the collector pump 1800-PP-504.
- Rougher flotation cell #3 collector via the collector pump 1800-PP-506.
- Primary cleaner column cell collector via the collector pump 1800-PP-508.
- Primary cleaner collector via the collector pump 1800-PP-510.

- Primary cleaner scavenger collector via the collector pump 1800-PP-512.
- Fine flake column cell collector via the collector pump 1800-PP-514.
- Fine flake cleaner collector via the collector pump 1800-PP-516.
- Fine flake cleaner scavenger collector via the collector pump 1800-PP-518.
- Attrition column cell collector via the collector pump 1800-PP-519.
- Attrition cleaner collector via the collector pump 1800-PP-520.
- Secondary attrition column cell frother via the collector pump 1800-PP-521.
- Secondary cleaner collector via the collector pump 1800-PP-522.

17.13 Froth Storage and Dosing

The frother storage isotainer 1800-TK-524 distributes the frother to the following equipment via the following pumps:

- Flash flotation frother via the frother pump 1800-PP-528.
- Rougher flotation cell #1 frother via the frother pump 1800-PP-530.
- Rougher flotation cell #3 frother via the frother pump 1800-PP-532.
- Primary cleaner column cell frother via the frother pump 1800-PP-534.
- Primary cleaner frother via the frother pump 1800-PP-536.
- Primary cleaner scavenger frother via the frother pump 1800-PP-538.
- Fine flake column cell frother via the frother pump 1800-PP-540.
- Fine flake cleaner frother via the frother pump 1800-PP-542.
- Fine flake cleaner scavenger frother via the frother pump 1800-PP-544.
- Attrition column cell frother via the frother pump 1800-PP-546.
- Attrition cleaner frother via the frother pump 1800-PP-548.
- Secondary attrition column cell frother via the frother pump 1800-PP-550.
- Secondary cleaner frother via the frother pump 1800-PP-552.

17.14 Flocculant Mixing and Dosing

The dry flocculant gets deposited into the concentrate flocculant hopper 1800-TK-574. The dry flocculant gets drawn via the concentrate flocculant hopper screw feeder 1800-FD-576. The flocculant from the feeder gets discharged into the concentrate wetting out head 1800-ZM-578, which discharges into the concentrate flocculant mixing tank 1800-TK-580. The concentrate flocculant mixing tank is equipped with an agitator 1800-AG-582. The mixed flocculant is therefore transferred to the concentrate flocculant dosing tank 1800-TK-584. The concentrate flocculant is therefore distributed to the final concentrate thickener and attrition concentrate thickener via 1800-PP-586 and 1800-PP-587 respectively.

The dry flocculant gets deposited into the tailings flocculant hopper 1800-TK-558. The dry flocculant gets drawn via the tailings flocculant hopper screw feeder 1800-FD-560. The flocculant from the feeder gets discharged into the tailings wetting out head 1800-ZM-562, which discharges into the tailings flocculant mixing tank 1800-TK-564. The tailings flocculant mixing tank is equipped with an agitator 1800-AG-566. The mixed flocculant is therefore transferred to the tailings flocculant dosing tank 1800-TK-568. The tailings flocculant gets pumped to the final tailings thickener via 1800-PP-570.

The flocculant spillage gets pumped to the final tailings thickener via the flocculant spillage pump 1800-PP-590. The flocculant mixing and dosing circuit is equipped with a flocculant/coagulant area safety shower.

17.15 Coagulant Mixing and Dosing

The dry coagulant gets deposited into the coagulant hopper 1800-TK-594. The dry coagulant gets drawn via the concentrate flocculant hopper screw feeder 1800-FD-596. The coagulant from the feeder gets discharged into the coagulant wetting out-head 1800-ZM-598, which discharges into the coagulant mixing tank 1800-TK-600. The coagulant mixing tank is equipped with an agitator 1800-AG-602. The mixed coagulant is therefore transferred to the coagulant dosing tank 1800-TK-584. The coagulant is therefore distributed to the final tailings thickener, final concentrate thickener and attrition concentrate thickener via 1800-PP-606, 1800-PP-612 and 1800-PP-615 respectively.

18 PROJECT INFRASTRUCTURE

18.1 Mining Infrastructure

18.1.1 Mining Office

The Phase 1 mining office is located on the shared plant and mine terrace and is a prefabricated containerised structure. The phase 1 building will be extended and added to cater for the increase in personnel. The building will provide office space for the mining personnel including the mine manager, technical services manager, geology personnel, survey personnel, maintenance engineers and mining support staff.

The mine office will be constructed using from modular containerised units, which will be shipped to site fully constructed and outfitted which will minimize any site work required.

18.1.2 Plant and LDV Workshop

The Phase 1 EMV workshop will be converted to the plant workshop and LDV workshop. It will cater for all maintenance and re-builds of the plant. Below is an illustrative layout view of the current EMV workshop. This workshop houses 125m^2 of storage space, 4 offices as well as the clinic. (Figure 78) below.

Figure 78: Typical illustrative layout of EMV Workshop

18.1.3 EMV Wash Bay

The EMV wash bay will be provided for the maintenance and cleaning of vehicles and plant and will service the mining fleet and the LDV fleet. The Phase 1 wash bay will become redundant as it is too small to accommodate the Phase 2 mining fleet. A new wash bay, capable of accommodating the phase to mining fleet has been provided for at the new Earth Moving Vehicle Workshop.

The Phase 1 wash bay could be kept as an additional wash bay servicing LDV's, this will provide the additional safety benefit of separating the LDV's from the mining fleet.

18.1.4 New EMV Workshop

To accommodate the new mining fleet a new EMV workshop is envisaged with the following dimensions 22.5m long, 25m wide and 15m tall at eaves level. This workshop will also be serviced with a 15t overhead crane.

18.1.5 ANFO Storage

The Ammonium Nitrate Fuel Oil ("ANFO") storage building is located to the north of the mine and plant site, a minimum of 500m away from all proposed and existing infrastructure, villages, product transport route, access road and the mining operation (pit). Access to the ANFO storage building is via an access road off the existing road which runs in an east-west direction past the proposed mine site and is located on an engineered terrace.

Access to the ANFO storage building will be controlled by the blasting Contractor who is appointed to undertake the blasting on site and will also be managed in conjunction with the local police services (Gendarmerie Nationale) in the area.

18.1.6 Explosives Magazine

The explosives magazine consists of a fully double fenced containerised storage facilities, fully enclosed by a retaining earth wall on an engineered terrace. The facility is sized to house 4 x 40'' containers. During Phase 1 of the Project only two storage containers where purchased, for Phase 2 of the Project an additional two storage containers are proposed.

Access to the explosive's magazine will be controlled by the blasting Contractor who is appointed to undertake the blasting on site and will also be managed in conjunction with the local police services (Gendarmerie Nationale) in the area.

18.1.7 Access Road to Explosives Magazine

The access road to the and the explosives magazine was installed as part Phase 1 of the Project on site and as such no additional work in this regard is envisaged.

18.1.8 Main Haul Roads

The main haul roads between the plant and mine terrace and between the pit and the RoM tip ramp will be installed as part of the construction works on site. The main haul roads have been designed to accommodate the mining fleet.

The haul roads are a minimum of 15m wide and accommodate 2-way traffic with no requirement for a berm in the middle of the haul road. A berm is included on the outer edge of the haul road, min 1.5m high.

18.1.9 ROM Tip Ramp

A second RoM tip ramp will be required for Phase 2 of the Project and has been included near the Phase 1 RoM Tip Ramp, to reduce the haul distance and additional hauling road.

The RoM tip wall will be constructed form reinforced earth and the trucks will dump directly into the primary crusher feed bin as appose to Phase 1 where the crusher feed bin is fed by a front-end loader.

18.2 Plant Infrastructure

The infrastructure to support the Phase 1 processing plant operation will be augmented and increased in size to accommodate the increased capacity.

The Plant infrastructure includes for:

- Plant Office – 4 times the size of the Phase 1 offices.
- Plant and LDV Workshop – converted from the Phase 1 EMV workshop.
- Laboratory.
- Process Water Dam.
- Raw Water Dam.
- Ablution Facilities.
- Reagent Stores.

18.2.1 Plant Office

The plant office has been increased to 480m^2 from the phase 1 size of 120m^2 and will house reception area, office boardroom, ablution facilities, server room, open plan office space and kitchenette.

18.2.2 Plant and LDV Workshop

As described under 18.1.2 the Phase 1 EMV workshop will be converted and re-purposed to fulfil this role.

18.2.3 Laboratory

The laboratory will be expanded by a further 3 containerised units to augment the 40" container installed under Phase 1.

18.3 Raw Water Supply

A dynamic water balance model was used to quantify water demands throughout the proposed LoM. The re-use of dirty water in mining processes was maximised to reduce raw water demands. The raw water will be sourced from a well field to be developed.

Phase 2 of the Project will require roughly 135m^3/hour of raw water, for this quantity of what a well field of approximately 20 boreholes will be required.

An environmental impact assessment will have to be undertaken for the well field to assess the impact and ensure that the necessary permitting can be obtained.

18.4 Process Plant, Storm Water and Raw Water Supply Dams

A Storm Water Management Plan ("SWMP") has been developed in accordance with IFC Environmental Health and Safety Guidelines for Mining (IFC, 2007) to mitigate potential water contamination and to ensure a safe working environment. The SWMP was set up to minimize and manage the loss of the water resource. The dirty water footprint is minimized and the use of dirty water in the ore beneficiation process is optimized.

The proposed SWMP measures include the implementation of Pollution Control Dams. Additional measures proposed include sediment traps where drains flow into the Pollution Control Dams to reduce the amount of silt that deposits in these dams.

The operational philosophy of mining operations will be to operate the process water system as a closed system which would not discharge into the environment. PCD water will, as far as possible, be used as a make-up water for dust suppression. The plant PCD will provide buffer storage capacity for this. Discharge from the PCDs will, however, occur and will need to be carefully controlled and monitored. Results from the salt balance, based on information from geochemical tests and modelling, suggests that PCD water under discharge conditions will comply with Madagascar and international criteria

The ranking of water sources to be used in the ore beneficiation process, in order of priority, are:

- PCD water from storm water / seepage from plant.

- Co-disposal, reject dump, ore stockpiles and open pit water).
- Raw water sources (Wellfield – Module 1). A raw water tanks will receive water from the well field system.

Both the process and storm water dam at the plant will be lined with a 2 mm HDPE liner to prevent water losses into the underlying ground and the liner will be installed on an A4 bidim geotextile protection layer.

18.4.1 Ablution Facilities

An ablution facility is provided for at the plant to accommodate the work force at the plant. It is envisaged that this facility will accommodate 120 people over and above the Phase 1 facility which caters for 30 people.

18.4.2 Reagent Stores

The reagents building is located adjacent to the plant and mine complex terrace. The building will serve as storage for the reagents required for the process plant. The building is open on all sides and will store barrels of flocculent and 220l drums of reagent.

18.5 Shared Infrastructure and Services

The shared infrastructure and services to support the processing plant operation and the mining operation was developed by Erustrat and positioned to optimise the overall mining and processing operation.

The shared infrastructure and services includes:

- Access roads.
- Terraces and bulk earthworks.
- Gate house and Turnstile access control.
- Re-fueling station.
- Waste water treatment plant.
- Potable water treatment plant.
- Storm water drainage.
- Storm water dams.
- Consumer sub-station.
- MCC's.
- Fencing.
- Fire water system, reticulation and storage.
- Potable water reticulation.
- Potable water storage tank.
- Waste water reticulation.

- Raw water supply.
- Strategic spares storage.
- Tire storage yard.
- General storage yard.
- Power generation facility.
- Fuel storage facility.

18.5.1 Access Roads

The access road had been developed as part of Phase 1 of the Project and currently on upgrading of the road is envisaged under Phase 2 of the Project.

18.5.2 Terraces and Bulk Earthworks

The main terraces for the works include:

- Permanent camp.
- Crushing circuit.

Terraces will be installed as part of the works and will be installed at a minimum of 300 mm above natural ground level to reduce the risk of flooding in the rainy season.

18.5.3 Gate House and Turnstile Access Control

On additional access control is envisaged for the plant. However, access control will be introduced between the change house and the process plant.

18.5.4 Re-fuelling Station

The re-fuelling station will be centralised on site and will be utilised by the plant and mine LDV fleet, and the mining haul truck fleet. The re-fuelling station is located on the plant terrace central to the site. The re-fuelling station will have a direct feed from the diesel storage facility on site and no provision has been included for any storage of diesel at the re-fuelling station.

18.5.5 Waste Water Treatment Plant

Provision has been included for the expansion of the waste water treatment plant installed during Phase 1 of the Project. The Phase 1 water treatment plant was specifically selected as a modularised plant to ensure that it could be easily expanded should the need arise. Waste water will be pumped from the mine site to the accommodation camp where the water will be treated.

The Phase 1 unit has been designed to accommodate the 70 people (2m^3/hour) with storage capacity of 210m^3 potable water. This system will be increased to accommodate an additional 280 people.

Drying beds have been included for the discharge and drying of the sludge from the units on the roofs of these units. These drying beds should be periodically cleaned.

18.5.6 Potable Water Treatment Plant

Provision has been included for the expansion potable water treatment plant installed during Phase 1. The Phase 1 water treatment plant was specifically selected as a modularised plant to ensure that it could be easily expanded should the need arise.

The Phase 1 unit has been designed to accommodate the 70 people ($2m^3$/hour) with storage capacity of 210 m^3 potable water, this system will be increased to accommodate an additional 280 people and the storage increased to 750 m^3 potable water. The unit is currently installed at the camp site and potable water will be pumped to the various locations required

18.5.7 Storm Water Drainage

Storm water run-off will be diverted around the plant mine terrace, the construction camp terrace, the open pit mine, ore stockpile areas, material reject dump and the co-disposal facility by a combination of unlined channels and berms which daylight downstream of the Project infrastructure. Material excavated from the unlined channels will be re-used to construct the berms adjacent to the channels.

Storm water falling on the plant and mine terrace, open pit, stockpile areas, material reject dump and the co-disposal facility is considered dirty water and as such, is collected in the storm water dams or Pollution Control Dams ("PCDs") located on site and discharge to the environment is limited. All the PCDs on site have been sized to accommodate a minimum of a 24 hour, 1:10 year return period storm event. Discharge from the PCDs will, however, occur and will need to be carefully controlled and monitored. Results from a daily salt balance model, based on information from geochemical tests and modelling, suggests that PCD water under discharge conditions will comply with Madagascar and international quality criteria.

18.5.8 Storm Water Dams

The storm water dam, or PCD located adjacent to the processing plant has been sized to accommodate all the storm water run-off from the plant and ore stockpile area. The dam has been sized to accommodate the 24 hour, 1:20 year return period storm event.

18.5.9 Consumer / Incomer Sub-station

The consumer sub-station will be a weatherproof prefabricated modular unit and installed on concrete bases and plinths with steel access stairways and landings. Access will be via the ends of the sub-station with one side having double door openings and pedestrian access on the other side. All guard railing to the landing on the access side will be removable to allow for equipment to be installed and removed as required throughout the duration of the mine's life.

The 2 x 1.4 MW, 3-ph, 400Vac diesel generators shall be controlled and connected to this substation, rated at 4,500A, 35kA either being stand-alone or part of the generator panels.

From this consumer sub-station, the power shall be distributed at 400Vac, via copper cables, to the 3-x respective plant MCCs, (Crusher, Mill and Floatation MCCs) located strategically in the plant, from where the plant loads and some infrastructure shall be supplied.

18.5.10 Plant Sub-stations

The plant sub-stations will be a pre-fabricated and installed on concrete bases and plinths with steel access stairways and landings. Access will be via the ends of the sub-station with one side having a service door and pedestrian access on the other side.

All guard railing to the landing on the access side will be removable to allow for equipment to be installed and removed as required throughout the duration of the mine's life.

The 3 plant sub-stations shall be powered form the consumer sub-station, each being of Form 1 separation, rated at minimum of 800A, 35kA, 3-ph, 400Vac and be of a Form – 1 isolation and construction format.

It is envisaged that the Crusher (MCC11), Mill and Floatation (MCC21 & 22), MCCs placed strategically to be closest to the loading, reducing in cable runs and installation costs.

18.5.11 Transformers

As the generators excite at 400Vac + N + E, no distribution transformers are required on this plant, other than that required by control and instrumentation.

18.5.12 Fencing

The Project site is located near the village of Fotadrevo (approx. 10 km by road) and smaller local villages (within 2 km of the site) and as such, the tailings storage facility, plant and mine complex, permanent camp and construction camp will be fenced. The security fence is intended to keep both unauthorised people and local livestock out of the areas detailed above.

The security at the plant and mine complex, the permanent and construction camp areas will be 2.4m high diamond mesh security fencing with 3 strands of galvanised barbed wire to the top of the fence. 6m wide access gates together with pedestrian access gates will be located at all the above-mentioned areas

Access control by means of booms, turnstiles and security check points will be installed at the various locations as required.

18.5.13 Fire Water System, Reticulation and Storage

The fire water system for the plant and mine area complex will be fed from the raw water dam located adjacent to the plant terrace. Sufficient capacity is included in the raw water tank to accommodate the fire water system requirements. Pump suctions will be installed such that the fire water requirements are not reduced / utilized except for firefighting purposes.

Fire hydrants and hose reels are proposed to be installed at the plant area only, with fire extinguishers (9 kg) being utilised at all the buildings / offices and the mining complex. The final layout of the fire reticulation system will be finalised once the preferred supplier is appointed during the execution phase of the Project.

18.5.14 Potable Water Reticulation

Potable water for the Project will be supplied via the potable water treatment plant which is fed from the raw tanks at the accommodation camp site. A well field will be installed which will provide all the raw water requirements for the Project, including all process, raw and potable water requirements.

The potable water reticulation system will include a network of buried uPVC pipes installed at the permanent camp and the plant and mine's complex. This network of pipes will be fed from the potable water supply storage tanks at the accommodation camp. Satellite storage tanks will be placed where required with local reticulation via pressure pumping.

Applications for water extraction permits and permitted water usage must be undertaken at execution stage by the Company with the assistance of GCS and Environment.

18.5.15 Potable Water Storage Tank

Provision has been made for an additional water storage capacity of 750m^3 at the accommodation camp site to augment the 230m^3 installed under Phase 1. This would equate to 14 days of water supply at 150 litres per person per day for 350 people.

The potable water storage will also supply the systems of the plant requiring potable / clean water like fire, gland seal water and the like.

18.5.16 Waste Water Reticulation

The waste water reticulation will comprise a buried network of uPVC pipes, manholes a waste water treatment plants which will collect all waste water from showers, urinals, kitchens, mess, basins, sinks, etc. and gravity feed this waste water to the waste water treatment plant. The waste water generated at the mine site will be pumped to the accommodation camp for treatment.

18.5.17 Strategic Spares Storage

A strategic spares storage area has been provided for which will support both the mining operation and the process plant operations. The storage area includes for two (2) 12m shipping containers which will be re-furbished and used as offices and storage.

18.5.18 Tyre Storage Yard

An open area has been identified on the plant terrace for the storage of spare tyres for the plant and mine LDV fleet and the mining fleet. Provision is included for security fencing at this location.

18.5.19 General Storage Yard

An open area has been identified on the plant terrace for a general storage yard which will be utilised to store oversize equipment and spares for the plant and mine operations that are not stored at the strategic spare's storage area. Provision is included for security fencing at this location.

18.5.20 Bulk Fuel Storage Facility

During the operational phase of the Project, the bulk fuel storage system will consist of five (5) containerised, double-walled fuel storage tanks, with sufficient storage capacity for two weeks operation (approx. 300,000 litres). Each storage tank has 60,000 litre capacity. The advantage of such a system is that very little is required in terms of civil and structural work to construct such a system, which in turn reduces capital cost.

During the construction phase of the Project, the fuel storage on site will have the capacity to store two weeks supply (approx. 120,000l) in two (2) containerised, double-walled fuel storage tanks. The tanks required at operation phase will be procured as part of the early works and mobilization of the Project and will be utilized during the construction phase of the Project. (Figure 79) below.

Figure 79: Typical Layout of Containerized Fuel Storage Tank

18.6 Power Supply - 150 tKpa E&CI

The total demand of the mine and plant averages to ± 59.4MWh per annum at an installed demand of 8.76MW running 24 hours, 7 days a week per annum at a diversity of 80%.

Due to the remote location of the Molo Graphite mine site, no other power, or electrical infrastructure is available to the site.

For the mine and plant it creates an opportunity to consider an alternative power supply, via an Independent Power Producer ("IPP") using renewable power sources such as thermal, solar, and/or batteries, or a combination thereof to be independent and not grid tied.

During the Molo-Phase I a diesel generation capacity of 3,108 kWe, as well as solar capacity of 2,592 kWp is being installed, plus a 1MVA / 1MWh Battery Energy Storage System ("BESS") as detailed in the existing IPP take-off agreement.

It is envisaged that the same approach shall be followed for Molo-Phase 2 relating to a diesel generation capacity of 8,305 kWe, as well as solar capacity of 8,908 kWp installed plus a 4MVA / 4MWh BESS.

The net effect of the above would result to a combined diesel generation capacity of 11,413 kWe, as well as solar capacity of 11,500 kWp installed.

18.7 Alternative Power Supply (IPP)

18.7.1 Plant, Mine and Infrastructure Power Supply

An 8,305 kWe thermal firm capacity is proposed that during the daytime when the solar (PV) 8,908 kWp is operating the BESS can assume a grid forming role ensuring stable supply to the mine at the same time allowing the diesel engines to be switched off to maximise the fuel savings.

The control system is configured to maximise the amount of excess power produced by the solar PV, to charge the battery. At no time does the diesel generators under normal operation provide power to the battery.

During the transition to the night time, as the solar resource decreases, the BESS will continue to discharge power while gradually bringing the diesel gensets into operation and to a level whereby the diesel gensets will resume the grid forming role and continue to take master control of power generation.

The control system is configured so that BESS delays any diesel gensets from starting if the BESS contains sufficient charge to ensure maximum thermal efficiency only as absolutely needed at night-time.

The BESS and controller are able to regulate the frequency and support voltage fluctuations during the starting and stopping of critical loads on the mine ensuring the internal power grid is maintained within acceptable tolerances and in balance, thereby reducing any unplanned power outages.

18.7.2 Plant, Mine and Infrastructure Power Supply Philosophy

The IPP sub-station shall consist of a:

- 8.305 Mwe prime rated diesel power plant.
- Bifacial single axis PV tracking plant to match load profile.
- BESS to match load profile and operating philosophy.
- Hybrid integrated energy control system.
- Face brick building control room single Point of Common Coupling (“PoCC”)
- Face brick building consumer sub-station.

From where it is distributed to the plant, mining load centres, as well as infrastructure, by means of PVC/SWA/PVC low voltage power and control cables, as well as MV-XLPE cable and/or 11 kV overhead lines.

Diesel fuel excluded.

Refer Figure 80 to the typical layout as indicated below.

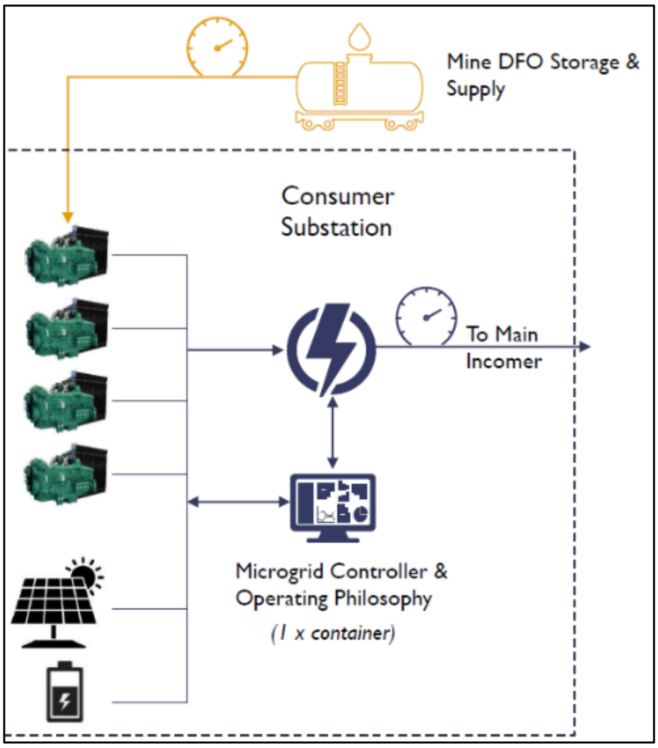


Figure 80: Alternative Power Arrangement

18.7.3 Thermal Design Philosophy

Taking into consideration that during Phase I, six Scania DC16-93A generators rated at 518 kW each were installed and this is to be complemented by the Phase 2, 5 x Cummins QSK60-G4 rated at 1.661 MW each.

It is envisaged that the total firm capacity shall be in the order of 11.412 Mwe, (prime rated power), all engines running.

The design philosophy is based on N+2 redundancy that guarantees availability even in the event of unexpected engine failure, (of any engine type) and major engine maintenance / overhaul.

Even with 2 of the larger engines out of service, this combined facility is capable of supplying the full site demand of 8 MWe.

Refer to Table 40 below indicating various operating and maintenance scenarios.

Table 40: Generation Operating and Maintenance Scenarios

Scenario	Operational Units	Total Firm Capacity
All engines available (Normal Operation)	6 x 518kW Scania 5 x 1661kW Cummins	11.412 Mwe
1 X Engine Unavailable	6 x 518kW Scania 4 x 1661kW Cummins	9.751 Mwe
2 X Engine Unavailable	6 x 518kW Scania 3 x 1661kW Cummins	8.090 Mwe

18.7.4 Solar Design Philosophy

The approach is that since solar PV provides the cheapest power, this source of power should be maximised.

By doing this typically the following are taken into consideration:

Technical restrictions are considered in configuring the solar PV facility:

- Solar PV in isolation is not grid forming and needs to synchronise with another grid forming device (e.g. grid, generators, or batteries).
- Solar PV power is not dispatchable and is only produced when the sun is shining.
- The generators will have to run at a minimum load factor requiring control and potential limitation of solar PV output.

In addition to the 2.5 MWp solar PV and 1MVA / 1MWh BESS of Phase I, the expansion of the renewable energy facility will imply a further 9 MWp of solar PV and 4MVA / 4MWh BESS.

Total of Phase I and Phase 2 renewable energy facility will consist of 11.5MWp solar PV and 5MVA / 5MWh BESS.

By integrating the BESS it is possible to stabilize the fluctuations of PV power and expand the solar PV facility to 11.5 MWp, off-setting more fuel by the cheaper solar power.

Overall the Solar PV Plant and BESS at an 11.5 MWp solar PV and the 4.8MVA BESS solution will result in ± 30% renewable energy contribution to the Phase 2 Molo Graphite Project.

18.7.5 Average Daily Generation

Figure 81 is a high level indication of how it is envisaged the Hybrid system will contribute to the daily power requirements.

This is the overall performance of the Alternative Power Supply inclusive of the Phase I, as well as Phase 2.

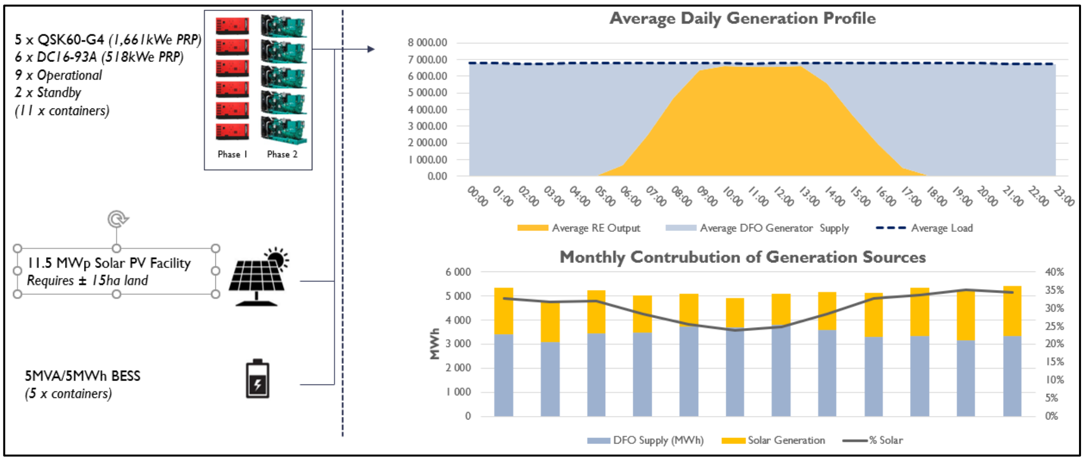


Figure 81: Average Generation Profiles

18.7.6 Real Estate

To install this Hybrid Alternative Energy Plant it is envisaged that additional real estate of 10Ha north of the Phase I facility is required.

This shall fall within the expansion of the RAP and be in the order of 400m x 250m as indicated in Figure 82 below.

Figure 82: Hybrid Alternative Energy Plant – Real Estate

18.8 Tariff Model and Carbon Savings

18.8.1 Expected Tariff

NextSource is currently bound in an IPP agreement with Cross Boundary Energy, who is delivering a dispatchable power solution with up to 33% renewable energy contribution, at a tariff of US$76.1/MWh (20 year agreement, excluding cost of fuel), for the Molo Graphite Phase I.

Based on this Cross Boundary Energy indication to deliver an overall renewable energy contribution for Molo Graphite Phase 2, 150 ktpa of up to 30% at a tariff of US$68.5/MWh on similar terms.

18.8.2 *Typical Cost Scenarios to be Expected*

The proposed solar hybrid solution for the Phase 2 expansion will result in over US$3M savings in year 1 with a reduction in CO_2 emissions of over 11,300 tonnes/year when compared to a thermal only solution.

The detail is contained in the Table 41 below.

Table 41: Cost Saving Scenarios

Variable	**Base Case Phase 2 (Thermal Only)**	**Phase 1 (Solar + BESS + Thermal**	**Phase 2 (Solar + BESS + Thermal)**
Installed Capacities			
Solar PV Capacity	NA	2,592 kWp	11,500 kWp
Diesel Generator Capacity	11,413 kWe	3,108 kWe	11,413 kWe
Operating Values			
Total Annual Site Consumption	59,420 MWh/year	11,212 MWh/year	59,420 MWh/year
Solar PV Contribution	NA	3,724 MWh/year	17,957 MWh/year
Solar PV Contribution	NA	33%	30%
Generator Contribution	59,420 MWh/year	7,488 MWh/year	41,463 MWh/year
Curtailed Solar Production	NA	NA	NA
Fuel Consumption	13,820,000 litre/year	1,728,019 litre/year	9,600,000 litre/year
Financial Analysis			
Total Power Plant Costs (CAPEX and O&M)	US$3,760,002/year	US$853,233/year	US$4,070,270/year
Fuel Costs*	US$11,056,000/year	US$1,382,415/year	US$7,680,000/year
“All in” Cost of Power	US$14,816,002/yr	US$2,235,649/year	US$11,750,270/year
Year 1 Net Savings	NA	US$537,395/year	US$3,065,732/year
Total Blended Cost of Power Estimate	US$249.3/MWh	US$199.4/MWh	US$199.7/MWh
Carbon Emissions	37,134 ton CO_2/yr	4,643 tons CO_2/year	25,795 tons CO_2/year
Carbon Savings	NA	2,275 tons CO_2/year	11,339 tons CO_2/year

*Assuming a fuel cost of US$0.8/litre

Note: Net savings calculations will, (where applicable), reflect deemed energy costs, reduced thermal energy usage / loading as a result of renewable penetration and may vary from headline values as a result of rounding.

18.8.3 Applicable Assumptions and Exclusions

In arriving to the abovementioned, various model assumptions were taken considered and should be validated during execution.

It excludes additional contingency that is normally attributed to a Project at this stage, unless there is a material change in load profile, technical specification, or any of the below assumptions, it is not expected that there will be any significant reduction in the pricing submitted.

No load profile data available due to Greenfield nature of the Project is available. The mine expansion load list and operating parameters were used to provide an estimated load profile with average load of 6.7MW and a max load of 8.1MWe. The load profile provided was assumed to be very stable of 75% of installed loads, with 12 hour and 24 hour maintenance downtime every alternative 14 days.

Average load profiles for accommodation, office buildings, ablution and workshops, as well as operation of the well fields for the expanded mine capacity were estimated based on experience of similar Projects and information for Phase I.

It is assumed that the well field pumps will feed from the main sub-station and will only be operated during daylight hours while solar PV generation is available.

Base diesel price of US$0.8/litre used in this proposal.

The thermal plant operating philosophy combined with the processing plant's expected power quality and controls will require generator minimum load factor of 40% and a minimum of 1 generator in operation.

Solcast data used as the solar resource in PVSyst simulation of the yield.

Site clearing and levelling (if required) is excluded.

18.8.4 Pricing Assumptions

Estimated PV module price for procurement in Q4 2022 = US$0.22/Watt.

The mine will consume at least 59,400 MWh per annum (from estimated load profile).

Ground conditions assumed to be suitable for standard piling to be confirmed following geotechnical investigation.

Connection point assumed to be capable of evacuating all the power from the hybrid power facility.

Terrain is assumed to be relatively flat and accessible.

NextSource Materials will provide Buyer's Enabling Works as agreed in Phase 1 to also support development and execution of Phase I.

18.8.5 Summary of Presented Tariff

Cross Boundary Energy is currently in an agreement with NextSource for delivering a dispatchable power solution with up to 33% Renewable Energy contribution, at a tariff of US$76.1/MWh (20 year agreement, excluding cost of fuel), for the Molo Graphite Phase I.

For the mine expansion (Molo Graphite Phase 2 - 150 ktpa), Cross Boundary Energy indicated that the can deliver an overall renewable energy contribution of up to 30% at a tariff of US$68.5/MWh on similar terms. (Table 42) below.

Table 42: Summary of Tariff

Phase 2 Pricing Summary (Conforming Sizing)	20 Year Contract
Down Payment (US$)	-
Contract Term (years)	20 years
(Thermal + Solar PV + BESS) Capacity Charge Tariff (US$/MWh)	68.50
Tariff inflation (% p.a.)	2.5%
Solar PV Size (MWp DC)	11.5
BESS Capacity (MVA / MWh)	5/5
Minimum Off-take (MWh p.a.)	59,400
Renewable Contribution	30%
Fuel Saved by RE Contribution (litre p.a.)	4,220,000

18.9 Electrical, Control and Instrumentation Design

18.9.1 General

All specifications referred to may be substituted by the national specifications of the supplier, subject to agreement by the Engineer and the Client.

Where general technical reference is made in a section it shall apply to all other relevant sections as well.

18.10 Standards, Acts and Regulations

The design, manufacture and testing of all electrical equipment shall comply with the latest applicable standard specifications and codes of practice issued by the N.R.S., SANS, IEC, BSI, or by other recognised National Standards authorities.

Any National standard shall meet the requirements of the comparable IEC standard and shall be approved by the Engineer.

Any deviations from specifications shall only be permitted with express written approval by the Engineer.

Where conflict may exist between various specifications, standard and codes of practice, etc., the most stringent requirement shall apply. Disputes shall be resolved in consultation with the Engineer and be subject to his approval.

All equipment offered shall also comply with the following Acts and the latest regulations thereof:

Mine Health and Safety Act (Act 29 of 1996) into which is included the Minerals Act.

Occupational Health and Safety Act (Act 85 of 1993).

18.11 Cable Racking

In the plant area cables shall be run on vertically mounted hot dip galvanised mild steel ladder type cable racks.

Cables outside the plant area may be directly buried in the ground, or installed overhead. Cognisance shall be taken of cable theft, in terms of the choice of 'buried' cables, or 'on rack' routes. Cables within plant areas may be strapped by means of PVC straps whereas all cables outside of plant areas exposed to sunlight shall be strapped by means of stainless steel straps.

Buried cables shall be in trenches as detailed on the drawings and shall have cable markers on the surface at 30m intervals inclusive of markers at direction changes. Tape markers are to be buried a minimum of 150 mm above the cable in the trench. Special markers on surface shall be installed indicating cable joints on buried cables. These markers are to indicate the size, type and number of cable.

A detailed services servitude plan is to be made and kept up to date. This plan will form part of the official hand-over documentation to the Client.

18.12 Electrical

18.12.1 System Voltages

All equipment shall be specified to meet the following standards:

Generator supply network.

380V, 3 phase, 3 wire, 50Hz.

Symmetrical 3 phase fault subject to generation and TBC during detail design.

Single phase fault (TBC during detail design).

11kV reticulation (Client specific):

- 11kV (+10 - 7.5%), 3 phase, 3 Wire, 50Hz.
- 21.5kA 3 Sec, Symmetrical 3 phase fault subject to generation and TBC during detail design.
- Single phase fault (TBC during detail design).

Fixed speed and VSD motors ≤ 300kW and welding socket outlets:

- 380V (±10%), 3 phase, 3 wire, 50Hz.
- Symmetrical 3 phase fault (design) – 30kA (1 sec).
- Single phase fault (design) 10A (10 sec) limited by generator-NER and NER on 11kV/380V transformer.

- Maximum size transformer is 1,600kVA, 11kV/380V, Dyn11.

Lighting, small power (services buildings distribution) and overhead cranes in process plant and process plant:

- 380Vac, 3 phase, 4 Wire, 50Hz.
- 220Vac, single phase, 3 Wire, 50Hz.
- Symmetrical 3 phase fault (design) – <10kA (1 sec).
- Single phase fault (design) as 3 phase (neutral solidly earthed).
- Subsidiary DB's shall be cascaded off the main DB and be rated at 5kA.
- Workshop welding socket, cranes and portable equipment will be 380Vac, 3 phase. air conditioning and compressors also included.
- Plant welding sockets 380Vac, 3 phase, 4 wire.

Portable equipment:

- 380Vac, 3 phase, 4 Wire, 50Hz, or 230V,single phase, 50Hz

Control voltage:

- 110Vac, L-N for MCC Contactor coils and MCC controls.

Instrumentation, Control systems, Network and PC's:

- 24Vdc for PLC DI/DO.
- Analogue 4-20mA, 24Vdc (Loop powered preferred).
- 110Vac, LNE, stabilized (UPS) supply for instrumentation supply. (24Vdc loop powered preferred).
- 110Vac, LN, stabilized (UPS) supply for PLCs.
- 230Vac Inline UPS supplies for process plant critical servers and network equipment.
- 230Vac standby UPS supplies for process plant SCADA, general office network equipment, servers and critical PCs.

18.12.2 MV Switchgear

Switchgear shall be manufactured and tested in accordance with IEC 298 (A.C. Metal enclosed switchgear).

Switchgear is to be designed for indoor installation.

Distribution circuit breakers to be operated by means of "umbilical cord", or from a remote control panel. Motor starter breakers shall be under PLC control.

Voltage transformers to be fitted to each incomer.

Protection relays shall be microprocessor based, programmable and selected to comply with the protection philosophy detailed later in this document.

Overhead line feeders shall be fitted with sensitive earth fault.

Cable "live" indicators shall be fitted to all incomer circuit breakers.

Harmonics to be kept within National grid supplier and NRS 048 limits.

Power factor correction shall be done at MV.

Energy metering to be done on all MV main feeders within the consumer sub-station.

Metering equipment shall only be installed on “incomer” circuit breakers. Power consumption information shall be gathered via the power management system.

The switchgear shall be designed for the maximum peak voltage occurring under fault conditions. All impulse testing shall comply with IEC 60 part 1 and 2 and shall apply to the switchgear with current and voltage transformers fitted.

18.12.3 Transformers

Plant distribution transformers (11/0.38kV) shall be ONAN free breathing, double wound, Dyn11 and fitted with conservator tanks, Buchholz relays and temperature indicators / trips.

Transformers of 300kVA and smaller shall not have Buchholtz relays fitted. Transformers shall be manufactured and tested in accordance with SABS 780. Maximum transformer size for 380Vac supplies shall be 1,600kVA. Transformers are to be rated to suit connected loadings. Transformer neutrals of the 380V star winding are to be connected via a NER to limit any earth fault currents to 10A on 380V (secondary) systems. All transformers supplying MCCs shall be fitted with bus trunking / armoured PVC / SWA / PVC multi cores that link the transformer to the MCC incomer panel.

Power, lighting and small power transformers shall be ONAN, sealed, double wound, Dyn11 and rated for 380Vac secondary. Transformers shall be manufactured and tested in accordance with SABS 780. Transformers shall be sized to cater for at least the installed capacity of the load. Transformer neutrals are to be solidly earthed.

Transformer bays shall be fully enclosed within a bunded area complete with an oil drain valve and stone aggregate (35 mm to 40 mm) with a front lockable gate, or removable panel with hinged personnel access doors. Minimum clearances of at least 1200 mm shall be kept around terminal boxes. The roof shall be removable for access to a transformer in case of replacement.

18.12.4 Fire Detection and Suppression

Fire detection and suppression shall be required for all MCC’s and all fire suppression equipment to be installed outside the MCC on a suitable concrete pad.

18.12.5 System Earthing

MV Reticulation:

- The system shall be earthed via NER’s in the generator yard.
- 380V Reticulation.
- 380/230V system shall have the neutral point solidly earthed.
- Lightning protection and equipment earth bonding.

- Detailed earthing and lightning protection design shall be carried out by a specialist earthing and lightning protection Contractor. The earthing values shall be in accordance with the following:
  - SANS 10199 (2004): The design and installation of earth electrodes (Edition 2).
  - SANS 10313 (2008): Protection against lightning – Physical damage to structures and life hazards (Edition 3).
  - SANS 1063 (2008): Earth rods, couplers and connections (Edition 3).
  - SANS 62305 – 1 (2007): Protection against lightning Part 1: General principles (Edition 1).
  - SANS 62305 – 2 (2007): Protection against lightning Part 2: Risk management (Edition 1).
  - SANS 62305 – 3 (2007): Protection against lightning Part 3: Physical damage to structures and life hazards (Edition 1).
  - SANS 62305 – 4 (2007): Protection against lightning Part 4: Electrical and electronic systems within structures (Edition 1).

In general, the following earth reading values shall apply:

- Electrical sub-station yards less than 1 Ohm
- General plant structures and conveyors less than 10 Ohm

Conveyor section inter-connecting earthing shall be done by galvanised, or aluminium, copper conductors.

An earth mat shall be established at each MV step down sub-station in accordance with SANS standards for design of earth electrodes.

All electrical equipment shall be earth bonded via the sub-station earth bar.

Lightning masts shall be earthed using copper rods to give an earth resistivity of a maximum of 5 ohm. The various earths shall be linked using buried 70 mm^2 bare copper earth wire.

Protection of buildings and structures against lightning strikes shall be in accordance with the recommendations of SANS 10313: 2008 (Physical damage to structures and life hazards).

All structures shall be earthed via a perimeter earth of 70 mm^2 bare copper earth wire, supplemented where necessary with earth rods to achieve an earth resistance of 10 ohms, or less in accordance with SANS standards for design of lightning protection systems.

Provision shall be made for a specialist contractor to carrying out an earth resistivity survey.

18.13 Motor Control Centres

MCC's shall comprise of free-standing steel switchboards with bottom cable entry to IP54. The board must be designed for a minimum fault level of 30kA at 380Vac. All MCC panels and distribution boards of which the fault level exceed 10kA shall be type tested, or partially type tested to SABS 60-439/61-439.

All starters shall be of a fixed pattern, fully compartmentalized construction. Starters shall incorporate moulded case circuit breakers (MCCBs), Contactors, electronic intelligent overloads for starters and earth leakage detection.

The MCCs design shall be for Type-2 co-ordination providing that the MCCBs are full fault rated as stand-alone devices and Contactors shall be rated for 1.2 X 106 operations at AC3 duty, or 60,000 operations at AC4 duty, or whichever is the more severe between this operation and Type-2 co-ordination. MCCBs shall be fitted with lockable door mounted operating handles.

MCCB fault rating shall comply with the maximum fault rating of the MCC panel and should be able to achieve this as a stand-alone unit.

The MCC shall comply with SANS 60-439/61-439 Parts 1 to 4 Low voltage switchgear and control assemblies.

All starters shall be wired for PLC control.

Variable Speed Drives shall be installed inside the MCCs, with a remote display module fitted onto the door. On VSDs greater than 75kW, an additional extraction fan must be fitted by the MCC manufacturer in order to extract the heat from the switchgear.

MCC buildings shall be fitted with industrial air conditioners for cooling. The temperature of the room shall be controllable from 20°C to 25°C. An industrial pressurization filter / fan assembly shall be fitted to the room to maintain a positive pressure inside. The filtration unit shall be self-cleaning and have an efficiency of 99% at 1 micron.

MCCs to have 20% spare allocation of cubicles.

18.13.1 Motors - 380VAC

380Vac motors shall generally be designed to SANS 60034-parts 1 and 2 and SANS 1804-parts 1 to 4. The duty type will be class S1 (continuous). Insulation shall be Class H. Temperature rise shall be limited to 80°C (Class B insulation). Enclosures shall be IP66 to IEC 144. Motors shall comply with EFF1 efficiency class (according to IEC 60034-30: 2008).

Motors sizes shall be as per the mechanical equipment list.

All motors larger than 230kW shall be on medium voltage.

Preferably WEG motors must be used in all electrical plant supplied for this Project.

18.13.2 Cables (Voltage Might Differ Between Sites.)

11kV Reticulation:

11kV cables shall be 3 core, copper conductor, XLPE/PVC/SWA/PVC, 11kV, Type A, individually screened type, manufactured and tested in accordance with SABS 1339: 2006 (Electric cables cross-linked polyethylene (XLPE) insulated cables for rated voltages 3,8/6,6kV to 19/33kV).

Standard “red stripe” cable is to be utilised for surface installations.

Transformer secondary cables at 11kV shall be multi core, copper conductor, XLPE/PVC/MDPE where the outer sheath is hardened for extra protection.

600/1000V Cables:

600/1000V cables shall be, copper conductor, PVC / PVC / SWA / PVC, 600 / 1000 volts manufactured and tested in accordance with SABS 1307. All cable on surface shall be of the "flame retardant" type standard "red stripe" cable. All power cables shall have a minimum of four (4) cores, the fourth core being used to provide a reliable earth connection to all drives.

Standard "red stripe" cable is to be utilised for surface installations.

Cable sizes shall be based on investigating the different sizes required by SANS 10142-1. The voltage drop shall incorporate the sizes of cables required during start-up conditions taking into account the voltage drop over cables with in-rush currents being in the order of 6 to 8 times the full load current. The most effective cable shall be chosen from the above requirements, taking into account the duty of the motor and the length of cable.

18.13.3 Lighting and Small Power

Lighting shall be by a combination of LED tubes, LED bulkhead and LED floodlights to achieve illumination levels as required by the Machinery and Occupational Safety Act as a minimum.

All lighting to be powered from suitable Lighting Uninterruptable Power Supplies (UPS's), with a minimum backup time of 30 minutes at full load.

The following lux levels re required on this Project:

Change Rooms	160 Lux
Plant equipment – General	130 Lux
Control Rooms	400 Lux
Outdoor Areas – General	30 Lux
Catwalks	30 Lux
Stairways	160 Lux
Conveyors	30 Lux
Sub-stations (indoor)	200 Lux
Sub-station Yards (outdoor)	30 Lux
Offices – Entrance Hall	200 Lux
Offices – Reception	300 Lux
Offices – General	330 Lux
Offices – Drawing	300 Lux
Offices – Passages	160 Lux
Workshops	300 Lux

High mast lighting shall be used in areas of high traffic of vehicles and for general lighting. The effects of the high mast lights on neighbouring farms shall be taken into account when placement and angle of lights are determined.

Switch sockets installed outdoor shall be enclosed with a drip cover. Dedicated switch sockets shall be installed in offices and control rooms where PC and networking equipment are reliant on UPS power.

18.13.4 Electrical Field Equipment

As an added point of isolating for emergencies, each drive shall have a local motor isolator with stop / start control station within a protective stainless steel drip cover to guard against physical damage. Field Isolators shall be fitted with an early break contact to ensure power to the main contactors are disconnected in order to prevent that the isolators are opened under “no load” conditions.

Conveyors shall be equipped with pull wire trip-switches on all accessible sides of the conveyors.

Belt slip switches shall be installed on all conveyors without belt scales. Belt alignment switches and belt rip and tear detectors shall be installed on all conveyors as detailed in the conveyor section.

All trip switches shall be hardwired to the conveyor starter panel. By activation of the trip switch will the conveyor drive motor/s be stopped immediately, with due indication on the SCADA, of the exact position of the activated trip switch.

Start-up audible sirens shall be installed for each conveyor at such intervals to provide sufficient warning at any point along the conveyor. Audible sirens shall be installed for starting-up warning of equipment per process area at least 10 seconds in duration

Allowance shall be made for the installation of 525V welding sockets in all plant areas, on positions as approved by the Engineer for maintenance purposes. These shall be installed in a protective steel drip cover to guard against physical damage.

18.13.5 Power Factor Correction (PFC)

If required it is to be done at MV, switching suitable rated PFC banks, in and out to maintain a unity power factor at the consumer sub-station.

18.14 Instrumentation Design and Drafting Software

Designs are done in MS-Excel, (or appropriate), as well as AutoCAD.

All specifications are done in Microsoft Word format.

18.15 Voltage Levels

The following voltages are applicable:

MCC based Discrete I/O linked to overload relays: 110VAC

Discrete I/O wired to PLC input, or output cards: 24V DC.

Analogue signals: 4-20mA. Loop power is preferred.

18.16 Field Instrumentation and Remote Input / Output (Rio) Equipment

Loop powered instrumentation should be used as far as possible. Density transmitters and magflow meters will require 110Vac external power supply if the loop powered option is not available.

Conventional instrumentation should be used, which implies that no "intelligent" instrumentation (e.g. DeviceNet type instruments) should be used except where specified explicitly. Profibus DP protocol will be used for communication to all density transmitters and magflow meters.

All belt-scales will have ControlNet interfaces. Instruments should be wired to junction boxes in the conventional manner.

Instrumentation, where required for accounting purposes, should have a high accuracy (± 0.5%), whereas those required for control purposes only, should have a lower accuracy and should be selected to meet the process requirements.

All field instruments requiring AC power should be supplied at 110V 50Hz derived from a dip-proofed UPS supply and should be supplied via instrument power distribution boxes.

The system design should be done using a hard-wired philosophy. In essence, field junction boxes should be used that are hard wired through to the MCC room using multi-core / pair cabling. All PLC IO cards should be mounted in marshalling panels installed in MCC / control rooms. This should be a climatically controlled environment.

Motor starter signals should be taken to the electronic overload relay. All stop signals from the E/stop should be hardwired back into the control circuit of the starter. The start signal should be an input into the electronic overload relay.

18.17 Instrument Cabling

Instrumentation cables should be Dekabon, (or equivalent) armoured. PVC/I/OAM PVC/APL/PVC cable should be used for all cables. Instrument cables should be individually and overall screened.

Red Stripe cable should be used in all cases.

As far as possible, instrumentation cable should be kept separate from power cables.

Where power and instrumentation cables are installed on the same rack, a minimum separation of 200 mm is required.

In general, instrument cables should be 1 or 1.5 square mm core diameter.

110V AC signals and 24V DC signals should not be combined in the same cable.

24V DC and analogue signals can be used in the same cable as long as each pair is screened.

18.18 PLC Panels and Junction Boxes

Unless otherwise specified the material of construction for panels and junction boxes should be powder coated mild steel sheet and/or rolled sections of not less than 1.5 mm in thickness.

The following colours should be used to powder coat the panels:

- Beige - All instrumentation and network panels, including the PLC panels in MCC's (in general, voltage level <= 110VAC).
- Orange (B26) - All electrical panels (In general, voltage level > 110VAC).

For panels installed in MCC's, the panels must be designed to provide a degree of protection of not less than IP54 as defined in SABS 1222.

Field mounted junction boxes must be protected to not less than IP65 standard. All junction boxes must have a canopy.

In additional to the above mentioned protection against the ingress of dust and water, it is essential that cable entries and openings are fully sealed against the ingress of vermin.

Where the design specifies that an MCC should incorporate marshalling / PLC compartments, all control and inter-locking circuits within each individual MCC must be wired through to and marshalled at a central point for connection to the master control and interlock cables. Marshalling compartments can be located at any convenient position in an MCC, but are to form an integral part of it. The height and depth of each such compartment should conform to those of the MCC, and the width should be not less than that of a single panel. However, the amount of equipment to be installed may be such that a width greater than that of a single panel may be needed.

Both front and rear access doors are to be provided and these are to be full height, fully gasketted and hinged on a vertical edge. Double doors may be used if the width of the compartment warrants it. All doors are to be secured by flush-fitting locks, with all such locks being operable with a common key.

18.19 Plant Control System

The plant control system will consist of a PLC and SCADA based system.

PLC Equipment:

The PLC's should be distributed throughout the plant and should be located in separate panels in the MCC building.

All motor control, valve control, interlocking, PID control and sequence start / stops should be controlled via the PLC control system.

18.20 Scada and Computer Systems

The plant control system will consist of a PLC and SCADA based system.

18.21 Networks

The PLC's and SCADA's will communicate over a TCP/IP Ethernet backbone network. Managed network switches will be used to minimize network collisions and to help guarantee the data throughput. This network will be configured in a redundant ring topology.

18.22 UPS Power

All Lighting, PLC's, SCADA's and network equipment should be powered from UPS power.

A single in-line UPS should be installed per MCC, or server room. In the MCC's, the UPS will also be used to dip-proof the control voltage.

A separated in-line lighting UPS shall be provided per segregated area as depicted by the detail design.

The UPS's should be double conversion On-line units with an inverter isolation transformer and static switch.

Provision should be made for a minimum of 30 minutes back-up time at 100% load. The UPS in the main server room should allow a minimum of 60 minutes back-up time at 100% load.

18.23 Level of Automation

All equipment must be controlled from the control room unless specified otherwise.

The automation design should be in accordance with the plant PID's.

Instrumentation and equipment selection should be in accordance with the Project approved Vendors list.

18.24 Control Modes

A three (3) level control mode will be implemented:

Maintenance

This mode is for maintenance only and all process interlocks will be by-passed. Individual drives can be started and stopped from the SCADA, or in the field. This mode is time limited and will revert back to interlock mode after a pre-set time has expired.

Interlock

The plant is controlled from the SCADA using the individual stop / start buttons on the SCADA. All process interlocks will still be active.

Sequence

In sequence mode the plant should be started and stopped using the PLC sequence logic. The field stop buttons are hardwired to the MCC and can be used in any mode.

All the modes will be PLC controlled.

18.25 Tailings Disposal and Storage Facility

Tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining and on the same basis as Molo phase 1. Costs have been factorised from the current Molo phase 1 estimate. In the next phase of the study a detailed design will be completed.

18.26 Geotechnical Report Assessment and Founding Strategy

A geotechnical investigation for the revised plant site and the camp site for the Molo Graphite Mine Project was conducted. The works included the excavation of 23 test pits, six DCP tests and the drilling of 11 rotary core boreholes. The fieldwork component for the camp site involved the mechanical excavation of 10 test pits and the advancing of five DCP tests.

The bedrock encountered in the test pits and boreholes was variable and ranged from extremely soft rock to hard rock strengths with variable weathering and jointing. The expected allowable bearing capacities for the varying hardness of Gneiss bedrock encountered across the site are summarised in Table 43 below.

Table 43: Expected Allowable Bearing Capacity for Encountered Gneiss Bedrock

Gneiss Bedrock	Allowable Bearing Capacity (kPa)
Completely weathered, foliated, Extremely Soft Rock	300
Very closely to moderately closely jointed, highly to slightly weathered, Very Soft Rock	500
Very closely to moderately closely jointed, highly to slightly weathered, Soft Rock	700
Very closely to closely jointed, medium to slightly weathered, Medium hard to Hard Rock	1000

As rock is located at a shallow depth at most locations it is recommended that structures generally be founded on rock rather than the overlying thin soils. However, light structures with loads of less than 100 kPa could be founded on the soils if necessary.

All temporary excavations in excess of 1.5m must be fully supported, or battered back to at least 1v:1.5h in soil and 2v:1h in rock. For excavations over 3.0m in depth, the soil and rock conditions should be assessed by a geotechnical engineer, or engineering geologist prior to and during excavation and the safe slope angles determined by analysis.

The transported and residual soils classify as G8 and G7 materials respectively and the very soft rock classifies as G6 to G7 class materials. These materials are however thin and of insufficient volumes to be considered as significant sources of construction materials for terrace construction.

The sandy lean CLAY ("CL") material presently approximately 6 km away from this site at the co-disposal site has an estimated available volume of 525 000m^3 of this material. The strength of these materials may be increased significantly with the addition of cement and/or lime and may then be suitable as terrace construction materials. Further laboratory testing will be required to determine this.

Due to the frequent rock outcrop and boulders present across the site, surfaces will need to be prepared by a bulldozer prior to trafficking.

Construction fill material for all terrace construction is sourced from the excavation for the return water dam. Approximately 450,000m^3 of material is to be excavated from the return water dam and approximately 383,000m^3 of this excavated material is to be crushed by means of mechanical jaw crushing. The total volume of material required for terraces is approximately 416,000m^3.

Concrete aggregate material for all concrete works on site will be sourced from borrow pit (s) on site, the material will be crushed using a cone crusher, supplied by the Company and operated by the earthworks and civil Contractor, to produce 19 mm concrete aggregate. The total volume of concrete aggregate is approximately 4,500m³ (5,800m³ of concrete).

19 MARKET STUDIES AND CONTRACTS

19.1 Graphite Overview

Graphite is highly refractory with a melting point of 3927°C, is chemically inert, is a superior lubricant and is one of the most electrically and thermally conductive non-metals. (Taylor, 2006; Moores et al., 2012; Olson, 2014). The carbon atoms in graphite are densely arranged in parallel stacked, hexagonal honeycomb lattice sheets, so graphite shows perfect basal cleavage (Olson, 2014). Graphite can be manufactured artificially in electric furnaces from petroleum coke, but is more cost effectively extracted from naturally occurring deposits (Taylor, 2006). Natural graphite is mined from three types of deposit:

1) Vein (also known as lump, or chip).

2) Amorphous.

3) Disseminated flake.

Benchmark Minerals Intelligence (Benchmark), 2020, estimated the overall natural graphite market to be 900 Mtpa, with the flake market accounting for 75%, amorphous accounting for 24% and vein accounting for less than 1% (Roskill 2020).

19.2 Flake Graphite Market

Natural flake graphite is used in industries requiring high temperature refractory materials, such as high temperature bricks and linings utilised in metal production, ceramics, petrochemical and cement industries (Benchmark, 2021). It is also used to dissipate heat in consumer electronics, as a gasket material for highly corrosive chemical environments, as a lubricant, as a liner in high performance brakes, as a carburizer in steel production and as a precursor for the production of graphene (Benchmark, 2021).

Natural flake graphite is mined predominantly in China, with over half of global supply coming from the Heilongjiang, Lubei, Shandong and Inner Mongolia provinces, (Benchmark, 2021).

Outside of China, Mozambique is the dominant flake graphite producer, followed by Brazil, Madagascar, India, Russia, Ukraine, Norway, Pakistan and Canada see Figure 83. Minor production also comes from Mexico, North Korea, Vietnam, Sri Lanka and Turkey (Statista 2020).

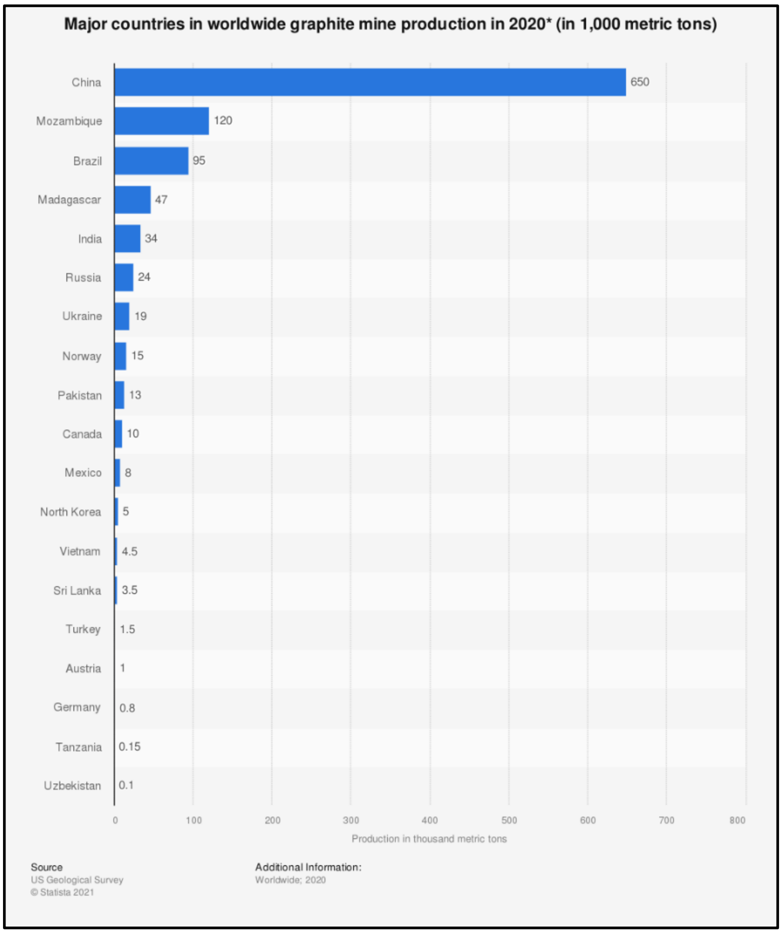


Figure 83 Natural Graphite Production Volumes by Country (Data from Statista, 2020)

Flake graphite has been identified as a critical and strategic material due to its essential applications in the aerospace and energy sectors, (Bloomberg New Energy Finance, (BNEF) 2021) and due to its role as the primary anode component in lithium-ion batteries (Benchmark, 2021). According to Benchmark (2021), the natural and synthetic graphite industries have little crossover in market share and end-uses with the exception of lithium-ion battery production where they are both used as anode material.

With the advent of the electrification of vehicles, it is forecast in the next 5 years that flake graphite's number one use will be in battery applications, overtaking its traditional industrial uses. As illustrated in Figure 84, the current natural flake graphite market is roughly 700,000 tpa with less than one third of that volume being utilized for battery anode material (Roskill, 2019).

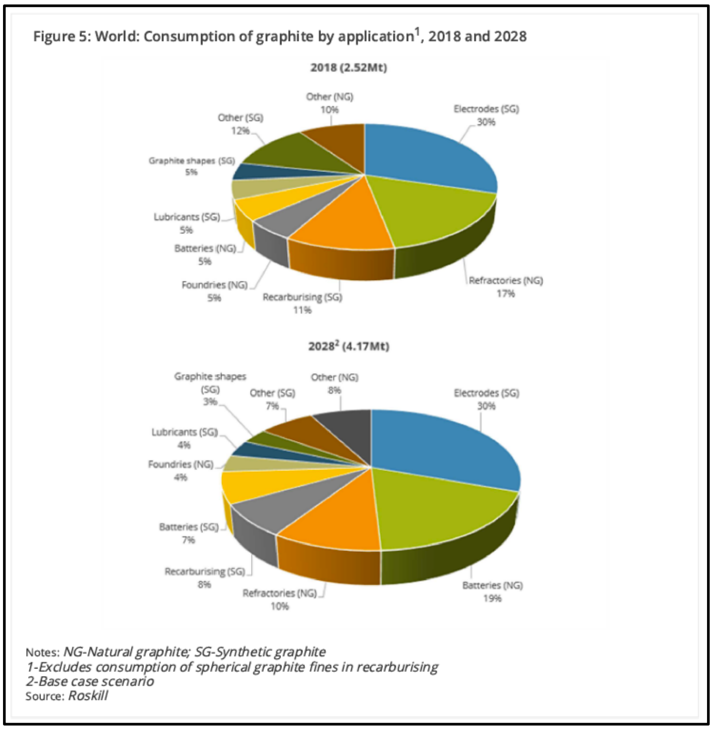


Figure 84: 2018 Estimated Production of 700,000 Tons of Natural Flake Graphite (Roskill, 2019)

Analysis by the International Energy Agency (2020) quantified that globally, the total number of Electric Vehicles (“EVs”) reached 7.2 million in 2019, with China accounting for 47% of sales. Currently, lithium-ion anode material represents almost 30% of the flake graphite market, but this number is expected to rise dramatically, with Benchmark (2021) projecting that total graphite demand will grow between 500% - 700% by 2030, with battery demand projected to account for 78% of total flake graphite consumption. The Organisation of the Petroleum Exporting Countries (“OPEC”) predicts EVs are set to approach 500 million by 2045, representing almost 20% of the global vehicle fleet.

Benchmark (2021) has predicted that the amount of flake graphite feedstock required to supply the lithium-ion anode market will grow from a current total of 120,000 Tpa over 4,000,000 tpa by 2025 (Figure 85).

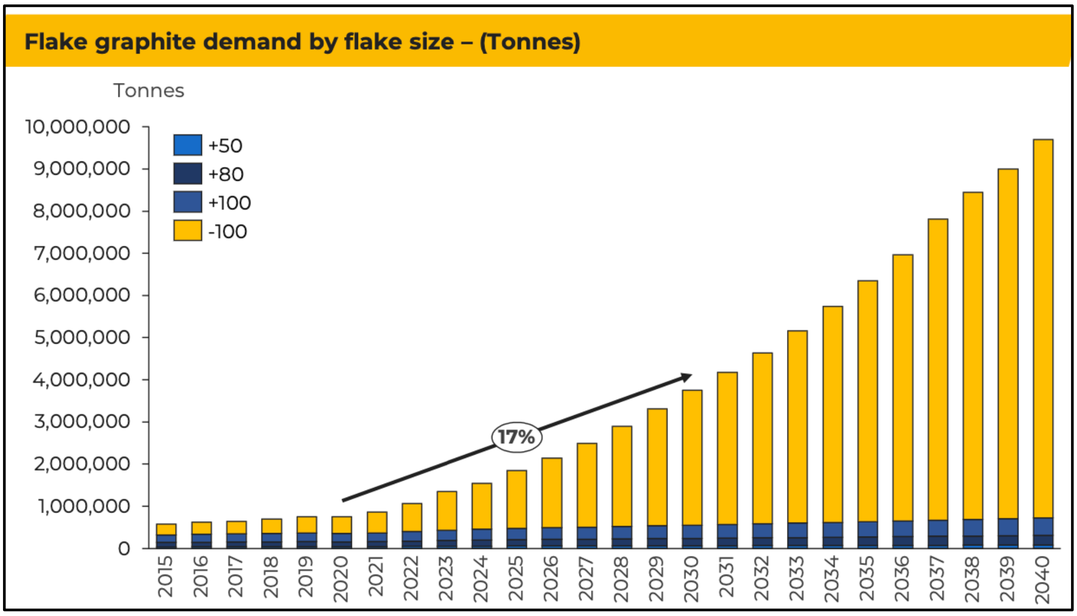


Figure 85: Flake Graphite Feedstock Required to Supply the Lithium-Ion Anode Market is Projected to Grow From 300,000 Tpa in 2020 to 9,000,000 Tpa by 2040 (Benchmark, 2021).

19.3 Flake Graphite Pricing

As an industrial mineral, flake graphite pricing is determined by three factors:

1) Flake size.

2) Carbon purity.

3) Industry-specific technical attributes of the flakes (Benchmark, 2021; Roskill, 2019).

Flake sizing is broadly classified into four ranges:

- Small (-100 mesh, or <75 µm).
- Medium (-80 to 100 mesh, or 75 µm to 180 µm).
- Large (-50 to 80 mesh, or 180 µm to 300 µm).
- Extra-large, or jumbo (+50 mesh, or >300 µm).

These flake sizes are in turn classified by carbon content (”C”) and are typically sold in ranges of 88% 93% C, 94% -95% C and 95% - 97% C. The specific technical attributes of the flakes are then defined by end-user parameters such as expansion coefficient, thermal and electrical conductivity and charge-discharge stability and efficiency. As the technical parameters sought by end-users are proprietary to their processes, pricing is not publicly available. There are, however, subscription pricing services that provide monthly graphite pricing for various flake sizes and carbon purities based upon input from graphite purchasers. Figure 86 identifies the average monthly flake graphite pricing for the past 12 months as provided by Benchmark (2021).

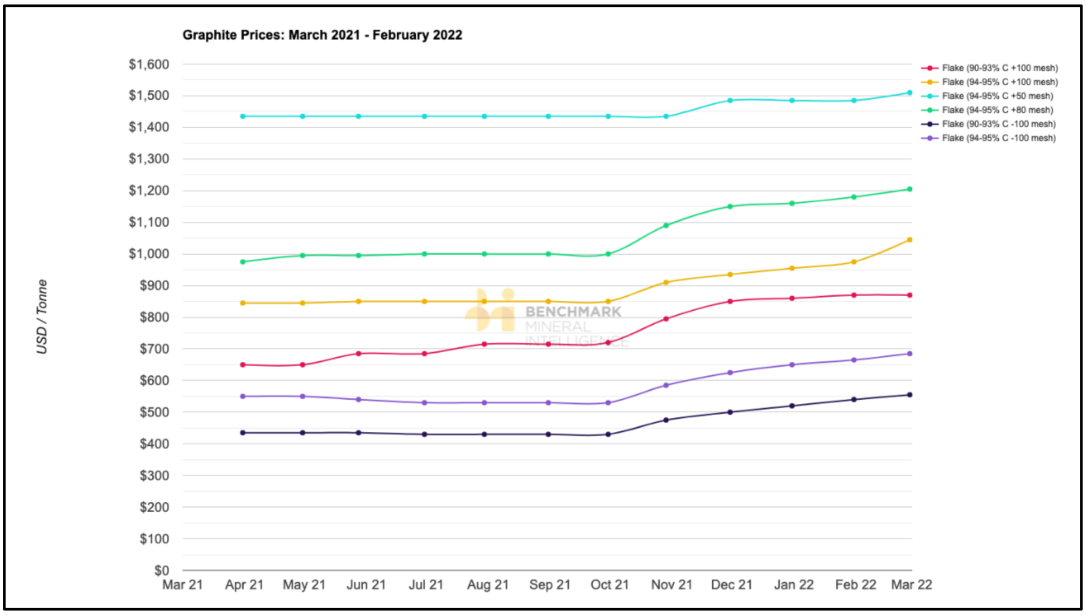


Figure 86: Monthly Flake Graphite Pricing for Various Flake Sizes and Carbon Contents (Benchmark, 2021)

Table 44 illustrates, the final flake graphite concentrate from the Molo deposit metallurgical work yielded material ranging from 96.9% C to 98.1% C.

Table 44 summarizes FOB China flake graphite pricing average from Benchmark and Fast markets (2022), over the past 12 months for material with a carbon content ranging between 96% - 97% C.

Table 44: 12 Month Flake Graphite Pricing With Carbon Contents Between 96-97% C and on A Fob China Basis

Date	+50 Mesh	+80 Mesh	+100 Mesh	-100 Mesh
31 Mar 2021	US$1,603	US$1,128	US$1,005	US$670
30 Apr 2021	US$1,603	US$1,173	US$1,005	US$670
28 May 2021	US$1,603	US$1,173	US$1,005	US$670
30 June 2021	US$1,603	US$1,173	US$1,005	US$648
30 July 2021	US$1,603	US$1,173	US$1,005	US$648
31 Aug 2021	US$1,676	US$1,173	US$1,005	US$648
30 Sept 2021	US$1,676	US$1,173	US$1,005	US$648
29 Oct 2021	US$1,676	US$1,374	US$1,139	US$771
30 Nov 2021	US$1,787	US$1,452	US$1,139	US$804
31 Dec 2021	US$1,787	US$1,452	US$1,162	US$838
31 Jan 2022	US$1,787	US$1,452	US$1,184	US$849
28 Feb 2022	US$1,843	US$1,508	US$1,340	US$894

Using the flake size distribution arrived at from metallurgical testing Table 25 with the average pricing as identified in Table 44, yields an average "basket price" of US$1,230.50 for Molo graphite as per Table 45 below.

Table 45: Current (January 2022) flake graphite pricing for Molo distribution

Microns	Mesh Size	C%	Yield (%)	Sale Price US$	US$/T
>300 µm	+50 mesh	96.9%	23.6%	US$1,787	US$421.74
180 µm – 180 µm	+80 mesh	97.0%	22.8%	US$1,452	US$331.05
150 µm – 180 µm	+100 mesh	97.2%	6.9%	US$1,184	US$81.69
<75 µm	-100 mesh	97.6%	46.7%	US$848	US$396.02
		97.2%	**100.0%**		**US$1,230.50**

19.4 Contracts and Agreements

Independent testing by various third-party end-users of flake graphite was announced by the Company in 2015, that confirmed that flake graphite concentrates from the Molo graphite mine meet, or exceed quality requirements for all major end-markets of natural flake graphite. The major end-markets for flake graphite include refractories, graphite anode materials used in lithium-ion batteries, specialty graphite foils used as essential components in the chemical, aeronautical and fire-retardant industries and graphene used in high-end ink and substrate applications.

The Company expects to sell most of the flake graphite produced at the Molo graphite mine through off-takes with several key customers. All of Phase 1 production is spoken for by off-take sales agreements by two main parties.

On October 16, 2018, the Company announced a binding off-take agreement for the supply of SuperFlake® graphite concentrate with a prominent Japanese Trading Company ("Japanese partner") that is a primary supplier of flake graphite to a major Japanese electric vehicle anode producer. To protect certain confidential aspects of the agreement, the Japanese Trading Company and the Japanese electric vehicle anode producer requested not to be identified. The key highlights are:

- Off-take is for a period of ten (10) years, beginning at the start of commercial production at the Molo Graphite Mine, with an automatic renewal for an additional five (5) years.
- Exclusive right to import and sell SuperFlake® graphite concentrate in Japan.
- Provided that commercial production commences within 3 years, following the ramp-up period, the Japanese Partner will purchase 20,000 tpa of SuperFlake® graphite.
- Product prices will be negotiated on a per order basis between the parties and will be based on the market prices (FOB basis) prevailing in the region.

On May 25, 2021, the Company announced that following a multi-year verification process, *thyssenkrupp* entered into a long-term partnership with NextSource and signed an off-take agreement to secure SuperFlake® graphite concentrate for their refractories / foundries, expandable graphite, (graphite foil) and battery anode production businesses. The key highlights are:

- Commercial agreement for the sale of 35,000 tpa of SuperFlake® graphite concentrate from the Molo mine.
- 10-year term with an automatic 5 year extension.
- Products under the agreement pertain to refractory, battery anode production and expandable graphite, (graphite foil) markets.
- Geographical regions include, but are not limited to, Europe, UK, North America, Mexico, China and South Korea.
- Minimum 7,300 tpa during Phase 1 initial production.
- Ramp up to 35,000 tpa in Phase 2.
- Shipments in Phase 1 will be used to verify run-of-mill production to trigger the larger volume expansion

On April 12, 2021, the Corporation announced a binding partnership agreement for the construction of a processing facility capable of converting flake graphite into spheronized and purified graphite ("SPG") and coated spheronized graphite ("CSPG") using the partner's proven processing technology. CSPG is sold to battery manufacturers where it is rolled to form the anode and is assembled with the cathode and other components into a finished lithium-ion battery. The Japanese and Chinese partners currently operate facilities that produce and sell SPG and CSPG to leading Japanese lithium-ion battery manufacturers that are part of the supply chains of Tesla and other major EV automotive companies. The Chinese partner will design and develop the process flowsheets, source all necessary processing equipment, and will provide all necessary training and operational know-how to produce SPG and CSPG material. In return, the Chinese partner will receive a 2% licensing royalty fee. The Japanese partner will leverage its sales relationships and will act as exclusive agent for sales, marketing, and trading of all SPG and CSPG material. In return, the Japanese partner will receive a 3% sales commission royalty.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

20.1 Description of Available Information

The location of this Project, specifically within a French speaking country and the common intricacies and nuances associated with Projects of this nature resulted in the establishment of a partnership between various South African based specialist Environmental consultancy firms and various a Malagasy specialist consultancy firms.

The availability of current and accurate environmental, social impact and permitting information on the Project is substantial. Table 46 provides a summary of the information which complements this report.

Table 46: Information Summary

Title	Author	Year

Environmental and Social Baseline (initial)	Agetipa	2011 - 2012
Environmental Legal Review	GCS	2013
Environmental and Social Sensitivity Study (spatially integrated)	GCS	2014
Final Terms of Reference (ToR) and Final Memorandum of Understanding (MOU) between ERG Madagascar SARLU (the Company) and the O.N.E. (regulator)	Agetipa and GCS	2017
Environmental and Social Impact Assessment and Management Plan (ESIA & ESMP) including residual baseline	GCS and Agetipa	2017
Relocation Action Plan (RAP)	Agetipa	2017
Conceptual Closure Plan	Globesight	2017
Permitting and Stakeholder Register	Environmental Law Group	2017
Revised and Finalise RAP for Phase 1	Harizo Rasolomanana	2021
Specific Environmental Management Plans	Harizo Rasolomanana	2021
Corporate Social Investment Plan (Phase 1)	Globesight	2021
Revegetation Plan (Phase 1)	Harizo Rasolomanana	2021
SD-HSSEQ Group Management System Developed	Globesight	2021
Erosion and Soil Management Plan	Harizo Rasolomanana	2021
Emergency Preparedness Plan	Harizo Rasolomanana	2021
Health Management Plan	Harizo Rasolomanana	2021

20.2 Applicable Laws and Standards

The FS and this PEA were undertaken in accordance with Malagasy national legislation, Equator Principles, World Bank and IFC requirements on environmental, cultural, health and safety protection.

20.3 Environmental and Social Sensitivities

The studies preceding the compilation of the ESIA revealed no fatal flaws, or land use restrictions in terms of environmental and/or social baseline conditions and all significant sensitivities were incorporated into the planning and design of the Project.

Various options were considered at commencement of the FS regarding the geographical placement of infrastructure, type and volume of water supply, type of electricity supply, tailings storage facility, (location and deposition strategy) and product transportation routes and shipping locations, specifically related to environmental and social benefits. Refer to Figure 87 for a depiction of the environmental sensitivities and final placement of infrastructure. The following final base case and alternatives have been selected and are relevant to this PEA:

20.3.1 Geographic Placement of Infrastructure

Sensitive areas were superimposed onto the Project design and the placement of infrastructure such as roads, processing plant, dams, Co-disposal, camp and other applicable ancillary infrastructure. Placement was undertaken in such a manner as to not have any significant impacts on the environment, or on the livelihoods of local inhabitants.

20.3.2 Water Supply

A detailed water study was initially completed by GCS Water and Environmental Consultants, and subsequently reviewed and revised by Geostratum.

20.3.3 Electricity Supply

Base case: Diesel generators.

Alternative: HFO (Heavy Fuel Oil) generators.

A trade-off study was undertaken to assess hybrids such as Diesel / HFO with Solar Photovoltaic (PV) and Diesel / HFO with Concentrated Solar Power (CSP), but none of these proved commercially feasible.

20.3.4 Tailings Storage Facility (TSF)

Tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining and on the same basis as Molo phase 1. In the next phase of the study a detailed design will be completed, complete with environmental and social impact assessment and closure.

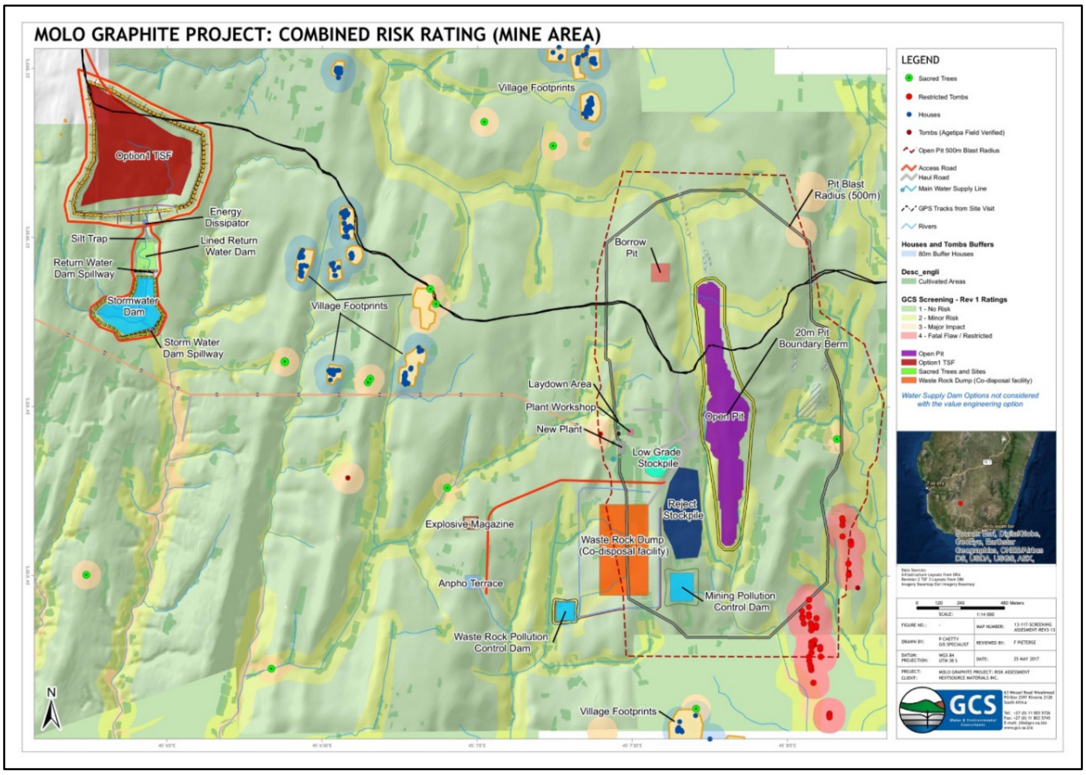


Figure 87: Infrastructure Overlaid on Environmental and Social Sensitivities

20.3.5 Transport Route and Shipping Port

The base case shipping port is Fort Dauphin via the RN10 and/or RN13. Refer to Figure 88.

Three shipping port sites were originally considered. These were Tulear, Soalara (Antseraky) and Fort Dauphin. Information obtained via the detailed Transport and Logistics Study indicated that weight limits on the Tongobory Bridge over the Unilahy River would render the transport of product to Tulear and Soalara (Antseraky) unviable. This route will in all likelihood be used to bring fuel to the proposed Project site. The only remaining feasible option is Fort Dauphin via the RN10 and/or RN13.

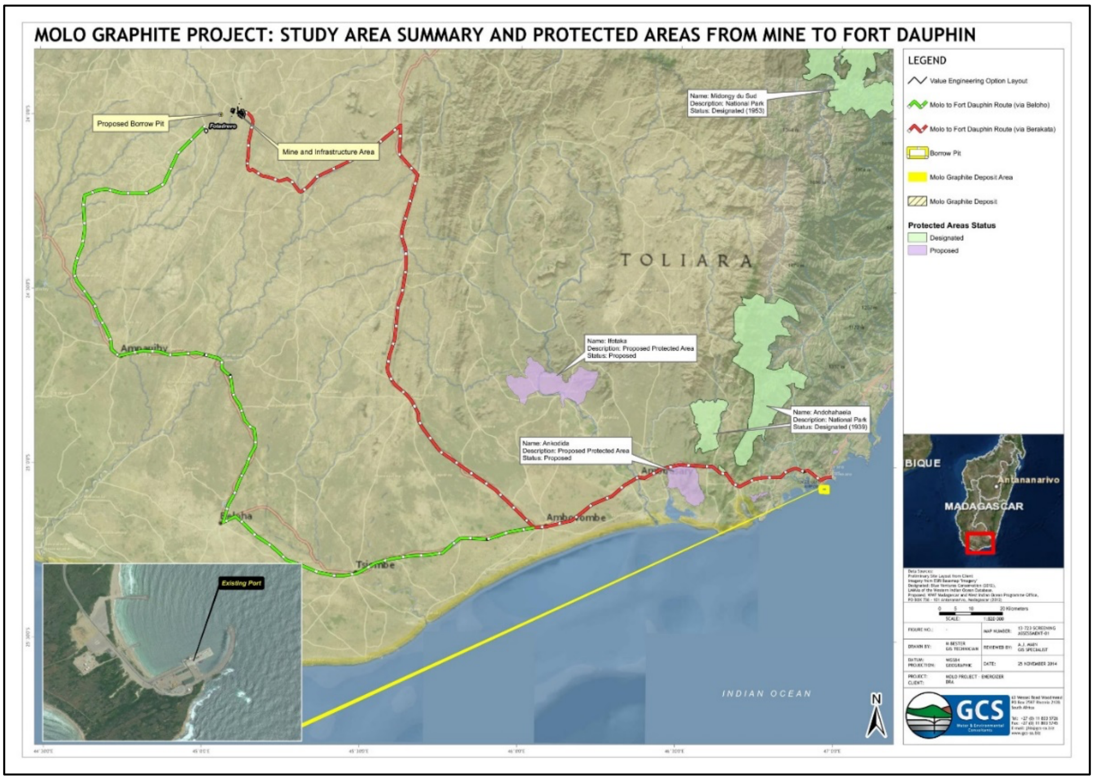


Figure 88: Transport Route Alternatives to Fort Dauphin

20.4 Environmental and Social Impact Assessment and Management Plans

Integrating knowledge of environmental and social sensitivities and risks into Project planning and design ensured that the final impact assessment component revealed that there are no fatal flaws from an environmental and social perspective. The significance levels of impacts range from minor to major before any mitigation measures are applied and from minor to average with mitigation measures included. Notably all major risks require significant reduction in risk via stringent controls. These controls have been incorporated into the Project design and planning with additional operational controls specified within the various environmental and social management plans.

To this end, the ESIA contains a chapter which details specific management measures which either remove the risks completely or reduce their significance to an acceptable level.

In addition, each specific environmental and social component has a prescribed monitoring plan which should be followed during each Project development phase. This is aimed at monitoring compliance against various specifications such as the baseline environment and predicted impact removal and reduction measures.

Post mitigation impacts are summarised as follows:

20.4.1 Preparation Phase

All potential impacts are rated from minor to average post mitigation. The preparation phase will give way to the construction phase, during which certain potential impacts will continue due to the nature of large-scale construction, noting that the potential impacts would be of short duration mostly. Soil compaction at and around the open pit area is seen as a potential average impact and the proposed mitigation measures should be adhered to.

20.5 Construction Phase

All potential impacts are rated from minor to average post mitigation. The most notable potential impact during this phase will be the construction of the waste rock / dry tailings co-disposal facility and potentially the TSF, which will have a visual change impact. The proposed mitigation measures are aimed at concurrent rehabilitation of the co-disposal and TSF, yet this can only commence once the construction phase gives way to the operational phase. Thus, the impact will be noticeable, yet of limited duration until such time as the side slope re-vegetation commences.

20.6 Operational Phase

The operational phase's potential impacts are rated from minor to average post mitigation. The most notable potential impact during this phase will again be the development of the waste rock / dry tailings co-disposal and the TSF, which will have a visual change impact. The proposed mitigation measures are to continue with the concurrent rehabilitation of the waste rock / dry tailings co-disposal and the TSF throughout the remaining life of the facilities. Final capping will occur during the closure phase. Thus, the impact will be noticeable yet of limited duration until such time as the new terraces are re-vegetated.

20.7 Decommissioning and Closure Phase

All potential impacts are rated from minor negative to minor positive post mitigation. A conceptual closure plan has been developed which indicates a positive, achievable and sustainable post closure land use. The post closure land use includes grazing land, agricultural development and managed domestic water supply. Active agriculture and managed domestic water supply are potential improvements on the base case, as these options to not exist at present.

From a conceptual closure perspective, the open pit will be left as a void upon closure. Difficulties could be experienced by concurrent in-filling in those cases where the ore body is limited to a single open pit, as is the case with the Project, and various grades of ore needs to be sourced from the pit. This requires access to the full pit and in-filling could sterilize ore reserves. In these cases, rehabilitation will be facilitated as follows:

The open pit perimeter walls will be rendered safe for humans and domestic animals. This is achieved by means of the following:

- Sloping the perimeter walls of the open-cast pit to the pit floor, or to stable ground water level that could establish within a reasonable period within the open-cast pit.
- Provide enviro-berms and ditches along the open-cast pit perimeter when perimeter wall flattening is not possible.
- Owing to removal of the mined product off site, and the creation of permanent mine residue facilities, insufficient material remains and a final void with respect to this Project is unavoidable.

- Conceptual land use options include a natural water body. Geochemical and hydrological studies indicate that the pit water quality will be suitable for stock watering and controlled irrigation (within limits).

These measures are in line with international practice and the opportunities available to utilise the flooded open pit for livestock watering and controlled irrigation provides a unique opportunity for the Project to contribute to a sustainable post mining land use.

Regarding the co-disposal facility, the minimum objectives for the closure and rehabilitation are to stabilise the facility both chemically and physically and thereby prevent air, ground, and water pollution, in accordance with the requirements of the relevant regulations and in line with good international practice. The intended end use considers the prior land-use and the location with respect to current and potential future socio-economic development. The walls of the co-disposal facility should be top soiled and vegetated progressively during construction and operation of the dam, whilst the upper surface may only be top soiled and vegetated once the final fill level has been reached.

20.8 Waste and Water Management

20.8.1 Mine Waste Assessment

A hydro-geochemical assessment was undertaken, including laboratory test work and modelling, on tailings material and representative waste rock. The results indicate that most sulphur associated with ore, tailings and waste rock consists of secondary sulphate minerals such as jarosite. The reaction rates associated with the secondary sulphate minerals are generally slow, resulting in relatively low sulphate loads in rainfall run-off and short-term operational seepage associated with the mine waste facilities. However higher sulphate loads could be expected in the long-term, specifically post-closure. The pH of water emanating from waste facilities will be neutral over the short term, decreasing to slightly acidic in the long-term. Metal concentrations will be low. Appropriate capping systems were specified to manage the long-term post closure water quality emanating from residual waste product.

20.8.2 Surface Water Environment

The Project site is located on the divide of three major river basins: Onilahy River Basin (32,361 km^2 catchment), Linta River Basin (6,177 km^2 catchment) and Menarandra River Basin (8,681 km^2 catchment). The calculated mean annual run-off is about 150 mm per annum, or roughly 18.8 % of the mean annual precipitation (MAP = 799 mm).

Around the Project site, all water courses drain either to the north, or south, away from the proposed mine infrastructure with most of the proposed mine infrastructure located in the locally southward draining Linta River Basin. Water courses are dry during the dry season and only flow during the rainy season.

Local surface water sources are largely under-developed, and no significant local water demands were identified during water use surveys.

20.8.3 Ground Water Environment

The Project site falls within the Ampanihy shear zone with a regional north-north-east – south-south-western trend. The local geology consists mainly of crystalline metamorphic rock such as quartz feldspathic gneiss. No significant faulting, or dyke intrusions have been identified with the site.

A number of hydrogeological boreholes were drilled and tested to investigate local aquifer conditions. The main aquifer zone is hosted within the weathered and fractures rock in the upper 50m below surface. The gneiss at the proposed mine site has low aquifer potential, with the marble layers to the west having a higher aquifer potential.

Community shallow hand-dug wells and boreholes are sparsely scatters over the area. Ground water use in the larger area is low, 15m^3 to 20m^3 per day maximum.

20.8.4 Water Management

The potential impacts that mining activities are likely to have on surface and groundwater resources in the surrounding environment were analysed. These potential impacts were quantified with different groundwater and surface water models and a significance rating was assigned.

Potential impacts relate to mine dewatering, wellfield abstraction, dirty water run-off, run-off reduction, and potential contaminant seepage from the co-disposal facility and material reject dump. None of the identified potential impacts posed a high, or serious risk to the downstream environment and the majority of risks were mitigated by the water management plan, which includes re-use of water, storm water management measures, water monitoring and waste capping systems.

20.9 Socio-Economics

A comprehensive assessment pertaining to the demographics and economic status of the communities in proximity to the Project was undertaken, specifically on those communities within, or adjacent to the area of influence. Significant reductions in the overall area of influence were achieved through careful consideration of environmental and social sensitivities to minimize the affected area.

Various avenues of engagement with stakeholders have been undertaken to date. These can be categorized as follows:

- Local and Regional engagement with government and other regulatory stakeholders.
- Public consultation.
- Land tenure and occupation.
- Cultural heritage.
- Sources of income.
- Engagement outcomes can be summarised as follows:
- Job creation.
- Education and skills transfer.
- Strengthening of municipal resources.
- Improvement in local security.

- Increasing business opportunities.
- Re-location of houses and associated assets, (minimum requirement in order to allow for public safety and operational efficiency).
- Loss of grazing land.

In this case though, there is a need to relocate crops and grazing land utilised by local inhabitants within close proximity to the Project site, for which a RAP has been developed.

20.10 Mine Rehabilitation and Closure

The development of a MRCP creates a platform for a mine to always have the end of the LoM in focus and to plan ahead for successful closure. Closure planning is guided internationally by different principles, frameworks, guidelines, and local legislation.

The framework for the MRCP specifically for the proposed Project was compiled from various recognised standards. These being:

- Equator Principals.
- International Finance Corporation ("IFC") Performance Standards.
- IFC Environmental, Health and Safety Guidelines for Mining.
- Madagascar Legislation.
- The International Council of Mining and Metals ("ICMM") guidelines.

The proposed Project is currently in the PEA Phase and requires, at this stage, a pre-conceptual MRCP. The conceptual MRCP was developed to reach the following target outcomes and goals:

- That future public health and safety are not compromised.
- The after-use of the site is beneficial and sustainable to the affected communities in the long term.
- Adverse socio-economic impacts are minimized and socio-economic benefits are maximized.
- Make the area of influence healthy and stable and restore its ability to allow another activity compatible with any form of life and activity in the region where it is located, after the closure of the mining operation.
- Ensure the safety of the location during and after the mining operation.
- Reduce the harmful effects of the mining operation on the atmosphere and on the water regime at an acceptable level.
- Integrate the mine and infrastructure landscape through appropriate negotiations and potential arrangements with the local municipality.
- Prevent the introduction of pests and weeds in areas where they were not present.
- Promote rapid re-generation and renewal of native plant species, or compatible with the ecosystem of the area.
- Re-habitation and closure estimates were annualised over the Project's LoM.

Refer to Section 21.1 for the closure estimates, based at end of LoM at present time value of money.

20.11 Permitting and Stakeholders

As input into the FS, pertinent information on the required permits / licenses to mine, including additional applicable permits / licenses for associated activities, was gathered and compiled into a permit register, a stakeholder database was compiled in support of the permitting register to assist with the identification of and approach to the relevant parties concerned.

The Madagascar Ministry of Environment's Office National pour l'Environnement (the National Office for the Environment), or "ONE", granted the Company its Global Environmental Permit / License for the 240 ktpa (Phase 1) Project on April 8 2019 after reviewing the following:

- Receipt of Exploitation Permit PE #39807.
- ESIA and RAP to IFC Performance and World Bank Standards.
- Completion of local and regional stakeholder and community engagement, with overwhelming support from both the local community and local government, as well as regional government.
- Signed agreements with all potentially affected land occupants to accept compensation for any affected crops and grazing land and relocation, if needed.
- Approved capital investment certification from the BCMM.
- Receipt of Cahier des Charges Minière (mining specification) from the BCMM as pre-requisite to submitting the ESIA and RAP to ONE for review.
- Successful completion of the ONE's technical evaluation process which consisted of a site visit and four separate community consultations
- Joint agreement and signature of the Cahier des Charges Environnementales (environmental specification) with the ONE.

In addition to the above a number of sectorial EMP's and other permits (listed below) for Phase 1 has either already been approved or are pending final approval. These are:

- Relocation Action Plan and Livelihood Restoration Plan for Phase 1.
- Thermal and Solar Self-Generation of Electricity for Phase 1.
- The Development of Roads and Pipelines for Phase 1.
- The Waste Management Plan for Phase 1.
- The approval of the following additional Specific Environmental Management Plans ("S/EMPs") is pending:
- The Development of the Base Camp.
- The Development of the Processing Plant and Associated Buildings and Structures.
- The Development of the Opencast Pit.
- The approval of the following additional permits is pending:

- The Tree Removal Permit.
- Industrial Operating License.
- Building / Construction Permit.
- Long-term Land Lease.
- Agreement with the Port of Ehoala.

21 CAPITAL AND OPERATING COSTS

21.1 Capex

The estimated Phase 2 construction costs are US$155,851,935 (including a 25% contingency). An additional US$20,876,922 is estimated for initial working capital. Over the LoM, an additional US$24,500,00M is estimated for sustaining capital and closure costs for equipment replacement and rehabilitation of the site at the end of the Project. The base date for the capital costs is January 2022 and no provision has been made for inflation. The accuracy of capital costs is considered to be within ±25%.

Table 48 summarizes the capital cost requirements.

Table 47: Capital Costs

Capital Cost Breakdown	Phase 2 Costs
Supply Items (Plant & Logistics)	US$43,200,996
Non supply Items (Engineering & Management, Civil & Infrastructure, P&G, Mechanical erection)	US$80,692,483
Sub Total	***US$123,893,479***
Fees and Contingencies	US$31,958,455
Construction CAPEX Total	**US$155,851,934**
Additional costs: Initial working capital Sustaining costs over LoM and closure costs at end of project	 US$20,876,922 US$24,500.000

21.1.1 Basis Of Estimate

The construction capital costs for the Project are built up as follows:

21.1.1.1 Modular Plant Capital Costs

The Phase 2 plant has been designed using the same modular approach as Phase 1 and the production capacity has been selected at an appropriate size that reflects current market realities.

The Phase 2 plant has been designed to a ±25% accuracy with the potential for variances during detailed engineering design and pricing, and from possible future COVID realities influencing items such as freedom of movement and its impact on logistics costs.

The same engineering and manufacturing company that was responsible for the Molo Phase 1 design and execution has provided a fixed price offer for the design, assembly, delivery and installation, complete with allowances for commissioning and spares. Table 48 summarizes the CAPEX breakdown of the Phase 2 processing plant.

Table 48: Phase 2 processing plant capital costs

Breakdown	Cost
Civils	US$1,124,000
Electrical & Instruments	US$2,913,000
Erection P&G Costs	US$4,707,000
Freight & Transport	US$6,613,000
Mechanical Equipment	US$24,035,000
Piping & Valves	US$2,059,000
Structural Steel	US$2,142,000
EPC	US$1,588,000
Contingency/Fees/Insurances	US$14,711,820
Sub-Total	**US$59,892,820**

21.1.1.2 Other Capital Costs

Table 49 summarize the Capex requirement for civil, earthwork, infrastructure, mining, EPCM and other capital costs.

Table 49: Other capital costs

Breakdown	Cost
Civils	US$20,017,344
Bulk Earthworks	US$3,374,888
Infrastructure	US$31,548,646
Mining	US$3,673,500
Buildings	US$964,235
Electrical & Instruments	US$5,750,000
Structural Steel	US$583,865
EPCM	US$12,800,000
Contingency/Fees/Insurances	US$17,246,635
Sub-Total	**US$95,959,114**

21.1.1.3 Mining costs

Mining capital costs have been developed from first principles using an owner mining philosophy where all equipment has been priced for supply and delivery to site. Operational costs are accurately captured based on first principles and expected organograms.

21.1.1.4 EPCM costs

These costs were built up based on the scope and time frame of the Project with an owners engineering philosophy. Phase 2 EPCM costs were estimated as 8% of the overall Capex less the performance bond and insurance.

21.1.1.5 Risk Allowances

A contingency amount of 25% has been allowed for Phase 2. This estimate is categorized as a preliminary economic assessment with a combination of market budget pricing and factorized pricing. The estimate is based on vendor quotations, Molo Phase 1 execution pricing, and market data.

21.1.2 Base Date

The base date for the Capital Cost Estimate is January 2022.

21.1.3 Base Currency

The estimate has been presented in the US$ currency in present day terms. Prices obtained in other currencies have been converted to US$ using the applicable exchange rates. Fluctuations in the exchange rates from the date of quotation to the date of indention have not been catered for.

21.1.4 Scope

The scope of the estimate covers the total capital cost of the plant areas and the related infrastructure as more fully described in the body of the main study documents. Owner's costs are all included.

21.1.5 Escalation (Inflation)

Inflation has not been considered in the Capital Cost Estimate.

21.1.6 Transport

Transport costs were included in the modular plant, as well as mining equipment costing where appropriate.

21.1.7 Overall Philosophy

The general approach to estimating was to measure / quantify each cost element from the PFD's, mechanical equipment list and infrastructure equipment list.

21.1.8 Assumptions

The estimate for the plant has been based on assumption of a continuous engineering, procurement, and construction effort with no interruption of the implementation program after funding approval has been obtained.

21.1.9 Spares, First Fill and Consumables

Allowances have been made for the first fill and for consumables based on the supplier and the process design engineer estimates.

21.2 OPEX

The average operating costs per tonne of graphite concentrate delivered on a FOB basis to the Port of Fort Dauphin, Madagascar, during the LoM is presented in Table 50. Certain Phase 1 fixed mining and G&A overhead costs will not increase because of increased production, which resulted in lower Phase 2 mining and G&A costs per tonne.

Table 50: Operating Costs Per Tonne of Graphite Concentrate

Breakdown	**Phase 2 OPEX**
Mining (US$/T)	US$145.88
Processing (US$/T)	US$206.75
Trucking to local port / Ft. Dauphin (US$/T)	US$133.00
General and Administration (US$/T)	US$10.00
Total	**US$495.62**

The reader should note that the estimated operating costs per tonne assume that the processing plant can successfully handle the variability in the ore body. As demonstrated by the SGS test work that is discussed in detail in Section 13, there is a risk that:

- The flake size distribution could be worse than expected.
- The product grade could be lower than expected.
- The recoveries could be lower than expected
- Combination of all of above.

If the plant does not perform as expected, this could have a material impact on operating costs.

21.2.1 Process Plant Operating Cost Summary

The average operating costs for the Phase 2 processing plant are presented in Table 51 below, which demonstrates that electrical power, reagents, and milling contribute the most to the plant operating costs at 43.9%, 16.4% and 18.8% respectively.

Table 51: Process Plant Opex Estimate

Description	Annual Cost US$/annum	Unit Cost US$/t milled	% Total
Power	13,481,546	5.39	41.9%
Water	13,176	0.01	0.0%
Reagents	5,569,675	2.23	17.3%
Milling	6,370,000	2.55	19.8%
Product Dryer	1,281,250	0.51	4.0%
Tailing	750,000	0.30	2.3%
Labour	2,206,847	0.88	6.9%
Maintenance	1,837,890	0.74	5.7%
Laboratory	525,303	0.21	1.6%
Miscellaneous Operating Cost	134,137	0.05	0.4%
Total Operating Cost	**32,169,824**	**12.87**	**100.0%**

21.2.1.1 Methodology

The processing plant operating cost (OPEX) estimate is based on the average steady state operating conditions of 2,500,000 tpa RoM feed rate and 150,000 tpa of final graphite concentrate product. Calculations to derive the estimate were informed by the process flowsheets, mass balance outputs, process design criteria and required mechanical equipment.

21.2.1.2 Labour

The staff organogram is presented in Figure 89 below. The staff headcount for Phase 2 is 79. The Process Manager (DU) will be an expatriate appointment while the other employees may be sourced locally. The organogram and staffing specifications were based on Erudite's experience in the operation of ore concentrator plants.

Labour cost breakdown is presented in Table 53 below. The labour cost was estimated at US$0.88 per tonne of ore milled.

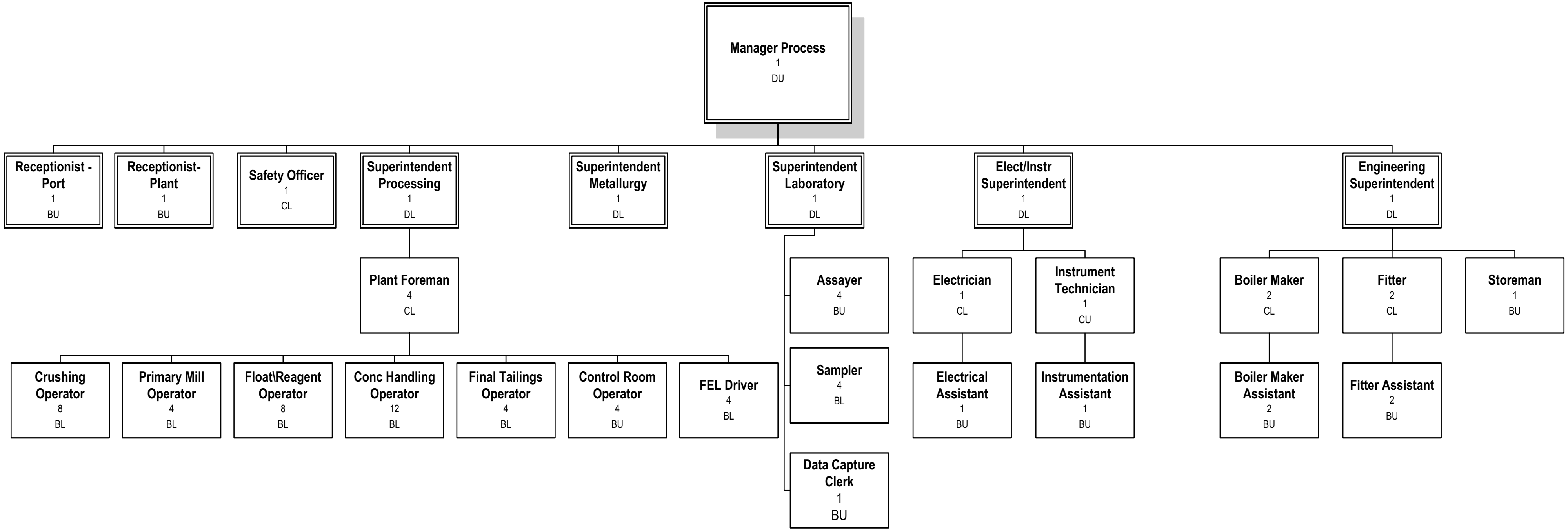


Figure 89: Molo 235 Mtpa Graphite Project Staff Organogram

Table 52: Labour Costs

Occupation	Site	Description	Grade	No.	Total	Total Monthly Salary US$	Total Monthly Medical Aid Cost US$	Total Monthly Transport Cost US$	Total Monthly PPE Cost	Monthly Cost US$	Annual Cost US$	Unit Cost US$/t Milled
Manager Process	Plant	Management	DU	1	1	10,327	1033	100	9	11469	265,816,8	0.1
Administrator / Receptionist	Plant	Management	BU	1	1	689	69	100	9	867	20,094,4	0.0
Administrator / Receptionist	Port	Management	BU	0	1	689	69	100	9	867	20,094,4	0.0
Elect/Instr Superintendent	Plant	Elect/Instr	DL	1	1	5,969	597	100	9	6675	154,706,3	0.1
Electrician	Plant	Elect/Instr	CL	1	1	1,165	117	100	9	1391	32,239,2	0.0
Instrument Technician	Plant	Elect/Instr	CU	1	1	1,936	194	100	9	2239	51,893,3	0.0
Electrical Assistant	Plant	Elect/Instr	BU	1	1	689	69	100	9	867	20,094,4	0.0
Instrumentation Assistant	Plant	Elect/Instr	BU	1	1	689	69	100	9	867	20,094,4	0.0
Engineering Superintendent	Plant	Engineering	DL	1	1	2,330	597	100	9	3036	70,365,3	0.0
Fitter	Plant	Engineering	CL	1	2	2,330	233	200	18	2781	64,455,2	0.0
Fitter	Port	Engineering	CL	0	0	-	-	-	-	-	-	-
Boiler Maker	Plant	Engineering	CL	1	2	2,330	233	200	18	2,781	64,455,2	0.0
Storeman	Plant	Engineering	BU	1	1	689	69	100	9	867	20,094,4	0.0
Boiler Maker Assistant	Plant	Engineering	BU	1	2	1,378	138	200	18	1,734	40,188,9	0.0
Filter Assistant	Plant	Engineering	BU	1	2	1,378	138	200	18	1,734	40,188,9	0.0
Superintendent Laboratory	Plant	Laboratory	DL	1	1	5,969	597	100	9	6,675	154,706,3	0.1
Sampler	Plant	Laboratory	BL	1	4	1,644	164	400	36	2,244	52,009,1	0.0

Occupation	Site	Description	Grade	No.	Total	Total Monthly Salary US$	Total Monthly Medical Aid Cost US$	Total Monthly Transport Cost US$	Total Monthly PPE Cost	Monthly Cost US$	Annual Cost US$	Unit Cost US$/t Milled
Sampler/Assayer	Port	Laboratory	BL	0	0	-	-	-	-	-	-	-
Assayer	Plant	Laboratory	BU	1	4	2,756	276	400	36	3,468	80,377,8	0.0
Data Capture Clerk	Plant	Laboratory	BU	1	1	689	69	100	9	867	20,094,4	0.0
Superintendent Processing	Plant	Operations	DL	1	1	5,969	597	100	9	6,675	154,706,3	0.1
Plant Foreman	Plant	Operations	CL	1	4	4,660	466	400	36	5,562	128,910,4	0.1
Plant Supervisor	Port	Operations	CL	0	0	-	-	-	-	-	-	-
Crushing Operator	Plant	Operations	BL	2	8	3,288	329	800	71	4,488	104,018,3	0.0
Primary Milling Operator	Plant	Operations	BL	1	4	1,644	164	400	36	2,244	52,009,1	0.0
Float/Reagents Operator	Plant	Operations	BL	2	8	3,288	329	800	71	4,488	104,018,3	0.0
Final Tailing Operator	Plant	Operations	BL	1	4	1,644	164	400	36	2,244	52,009,1	0.0
Concentrate Handling Operator	Port	Operations	BL	3	12	4,932	493	1,2	107	5,533,2	128,242,9	0.1
Control Room Operator	Plant	Operations	BL	1	4	1,644	164	400	36	2,244	52,009,1	0.0
FEL Driver	Plant	Operations	BL	1	4	1,644	164	400	36	2,244	52,009,1	0.0
Superintendent Metallurgy	Plant	R&D	DL	1	1	5,969	597	100	9	6,675	154,706,3	0.1
Safety Officer	Plant	Safety	CL	1	1	1,165	117	100	9	1,391	32,239,2	0.0
Total				**32**	**79**	**7,9493**	**8315**	**6701,2**	**708**	**95,217,2**	**2,206,847**	**0.9**

21.2.1.3 Power

The breakdown of the power cost is shown in Table 53 below. The power cost was estimated at US$5.35 per tonne of ore milled.

The power cost calculations are based on a blended power solution consisting of thermal generators, solar field and battery farm from the same IPP as is being used for Phase 1 at a fixed cost of US$0.0685 per kWh. The estimated 9,600,000 litres per year of diesel required for the generators will be supplied to the IPP by the Company at an average cost US$0.90 per litre resulting in a variable cost of US$0.1222 per KWh. The total blended cost was therefore estimated at $0.1907 per KWh.

Table 53: Breakdown of the Operating Cost for Power

Crushing		
Installed Capacity – Crushing	1,697.4	kW
Crushing Unit Power Cost	0.1907	US$/kWh
Crushing Power Consumption	9,664,996	kWh/year
Crushing Total Variable Power Cost	1,843,520	US$/year
Crushing Total Variable Power Cost	0.74	US$/t milled
Mill/Float/Tailings		
Installed Capacity - Mill/Float/Tailings	8,045.7	kW
Mill/Float/Tailings Unit Power Cost	0.1894	US$/kWh
Mill/Float/Tailings Power Consumption	45,812,216	kWh/year
Mill/Float/Tailings Total Variable Power Cost	8,738,311	US$/year
Mill/Float/Tailings Total Variable Power Cost	3.50	US$/t milled
Drying/Screen/Bagging		
Installed Capacity - Drying/Screen/Bagging	2,669	kW
Drying/Screen/Bagging Unit Power Cost	0.1894	US$/kWh
Drying/Screen/Bagging Power Consumption	15,202,297	kWh/year
Drying/Screen/Bagging Total Variable Power Cost	2,899,716	US$/year
Drying/Screen/Bagging Total Variable Power Cost	1.16	US$/t milled
Total Power Costs		
Annual Power Cost	13,481,546	US$/year
Monthly Power Cost	1,123,462	US$/year
Power Total Unit Cost	5.39	US$/t milled

21.2.1.4 Water

The operating costs for water were based on potable water usage of 8 784m^3/year at a unit cost of US$1.5/m^3. Table 54 below shows the water operating costs.

Table 54: Operating Costs for Water

Potable Water Cost		
Potable Water Cost	1.5	US$/$m^3$
Potable Water Consumption	8,784	m^3/year
Potable Water Cost	13,176	US$/year
Potable Water Cost	0.0053	US$/t milled
Total Water Costs		
Annual Water Cost	13,176	US$/year
Monthly Water Cost	1,098	US$/month
Water Total Unit Cost	0.0053	US$/t milled

21.2.1.5 Reagent Costs

The consumption figures used in determining the reagent operating cost estimate were based on the process design criteria information and are supported by the plant mass balance. The reagent costs used were based on imported supply rates obtained from reagent suppliers FOB: Faux Cap, Madagascar. Table 55 shows the reagent operating costs.

Table 55: Reagent Operating Costs

Collector		
Cost	890	US$/t
Dosage Rate	111	g/t
Consumption	277.5	t/year
Annual Cost	246,975	US$/year
Monthly Cost	20,581	US$/month
Unit Cost	0.099	US$/t milled
Frother		
Cost	4,340	US$/t
Dosage Rate	165	g/t
Consumption	412.5	t/year
Annual Cost	1,790,250	US$/year
Monthly Cost	149,188	US$/month
Unit Cost	0.716	US$/t milled
Flocculant (Concentrate)		

Cost	2,930	US$/t
Dosage Rate	1	g/t
Consumption	2.5	t/year
Annual Cost	7,325	US$/year
Monthly Cost	610	US$/month
Unit Cost	0.003	US$/t milled
Flocculant (Tailings)		
Cost	2,930	US$/t
Dosage Rate	125	g/t
Consumption	312.5	t/year
Annual Cost	915,625	US$/year
Monthly Cost	76,302	US$/month
Unit Cost	0.366	US$/t milled
Coagulant		
Cost	6,140	US$/t
Dosage Rate	170	g/t
Consumption	425	t/year
Annual Cost	2,609,500	US$/year
Monthly Cost	217,458	US$/month
Unit Cost	1.044	US$/t milled
Total Reagents		
Annual Reagent Cost	5,569,675	US$/year
Monthly Reagent Cost	464,140	US$/month
Reagent Total Unit Cost	2.23	US$/t milled

21.2.1.6 Milling

The milling costs shown in Table 56 consist of the grinding media costs, as well as the costs for re-lining.

Table 56: Milling Costs

Milling		
Balls Unit Cost	2.24	US$/t
Liner Unit Cost	0.308	US$/t
Balls Annual Cost	5,600,000	US$/year
Liner Annual Cost	770,000	US$/year
Total Annual Cost	6,370,000	US$/year
Total Unit Cost	2.55	US$/t milled

21.2.1.7 Product Drying

The cost of product drying shown in Table 57.

Table 57: Product Drying Costs

Product Drying		
Product Drying Cost	0.513	US$/t
Annual Cost	1,281,250	US$/year
Monthly Cost	106,771	US$/month
Unit Cost	0.513	US$/t milled

21.2.1.8 Final Tailings Handling

Table 58 shows the cost of tailings handling.

Table 58: Final Tailings Handling Costs

Final Tailings Handling		
Final Tailings Handling Cost	0.30	US$/t
Annual Cost	750,000	US$/year
Monthly Cost	62,500	US$/month
Unit Cost	0.30	US$/t milled

21.2.1.9 Maintenance Costs

The maintenance costs in Table 59 were estimated according to the capital cost of the mechanical equipment for the crusher, wet plant and dry circuits. The cost was estimated to be 7.5% of the mechanical equipment cost for each circuit.

Table 59:Maintenance Costs

Crusher		
Capital Equipment Cost	2,584,000	US$/t
Spares Allowance	7.5%	
Annual Cost	193,800	US$/year
Monthly Cost	16,150	US$/month
Unit Cost	0.078	US$/t milled
Wet Plant		
Capital Equipment Cost	17,182,000	US$/t
Spares Allowance	7,0%	
Annual Cost	1,202,740	US$/year
Monthly Cost	100,228	US$/month
Unit Cost	0.481	US$/t milled
Dry Plant		
Capital Equipment Cost	8,827,000	US$/t
Spares Allowance	5.0%	
Annual Cost	441,350	US$/year
Monthly Cost	36,779	US$/month
Unit Cost	0.177	US$/t milled
Total Maintenance Costs		
Annual Cost	1,837,890	US$/year
Monthly Cost	153,158	US$/month
Total Unit Cost	0.74	US$/t milled

21.2.1.10 Laboratory Costs

Table 60 below presents the breakdown of the laboratory costs for the Project. The estimated cost to process a single sample is US$1.50. A total of 157,341 samples is expected to be processed on an annual basis.

Table 60: Laboratory Cost Breakdown

Laboratory Plant Sample		
No of Samples	54,750	Per year
Cost per sample	1.50	US$/t
Annual Cost	82,125	US$/year
Monthly Cost	6,844	US$/month
Unit Cost	0.033	US$/t milled
Grade Control Samples		
No of Samples	81,162	Per year
Cost per sample	1.50	US$/t
Annual Cost	121,743	US$/year
Monthly Cost	10,145	US$/month
Unit Cost	0.049	US$/t milled
Concentrate Sample		
No of Samples	21,429	Per year
Cost per sample	1.5	US$/t
Annual Cost	321,435	US$/year
Monthly Cost	26,786	US$/month
Unit Cost	0.129	US$/t milled
Total Laboratory Costs		
Annual Cost	525,303	US$/year
Monthly Cost	43,775	US$/month
Total Unit Cost	0.21	US$/t milled

21.2.1.11 Miscellaneous

The miscellaneous costs Table 61 includes stationery and office equipment, lubricants, workshop tools and computer software. The costs were based on Met63 process plant experience and were discussed with the Company.

Table 61: Miscellaneous Costs

Description	Monthly Cost US$	Annual Cost US$	Unit Cost US$/t milled
Lubricant	6,879	82,545	0.035
Workshop Tools	1,720	20,636.4	0.009
Computer Software	2,580	30,954.6	0.013
Total Costs	11,178	134,137	0.05

22 ECONOMIC ANALYSIS

22.1 General

The economic analysis is based on the results of a cash flow financial model prepared in Microsoft Excel by NextSource. The financial model assumes a 12-to-18-month construction period and that the Phase 2 processing plant is built adjacent to the Phase 1 processing plant.

Inputs to the financial model were obtained from the various technical experts where possible, or from third parties considered experts in their field. All financial values in this report are based in United States Dollars (US$) and nominal terms. Values shown are rounded to an appropriate level.

The initial capital costs are assumed to be incurred on day 1 of the financial model and all other cash flows are assumed to occur on the last day of the respective year.

The results of the financial model are presented on a *pre-tax and royalties* and on a *post-tax and royalties* basis. No provisions were made for any debt financing of initial capital costs.

22.2 Assumptions

22.2.1 Product Price

The product price is based on current (February 2022) for +48, +80, +100 and -100 mesh sizes, resulting in a weighted average “basket” price of US$1,230.50 per tonne of flake graphite concentrate. The pricing information was provided by UK-based commodity price reporting agencies Benchmark Minerals Intelligence and fast markets. No inflation factors were applied to the product price during the LoM resulting in a constant price of US$1,230.50 per tonne over the LoM.

22.2.2 Mineral Royalties

The Madagascar government retains a 2% gross revenue royalty, Vision Blue retains a 3% gross revenue royalty, and Malagasy retains a 1.5% net smelter return royalty on the Project.

22.2.3 Technical Assumptions

The primary technical assumptions used in the financial analyses are presented in Table 62 below.

Table 62: Scenario 2 Technical Assumptions for Financial Analysis

Item	Assumption
Average ore mined per year (LoM) after year 2	2,532,345 tpa
Average stripping ratio (waste to ore ratio)	0.53:1
Life of Project	27 years
Life of Mine	26 years
Average mill feed/head grade (% C)	6.16%
Average concentrate grade (% C)	97.3%
Average mill recovery (%)	88.3%
Annual average production of graphite concentrate (LoM)	150,000 tpa

22.2.4 Economic Assumptions

The primary economic assumptions used in the financial analyses are presented in Table 63 below

Table 63: Economic Assumptions for Financial Analysis

Item	Assumption
Currency	US$
Average graphite price	US$1,230.50/T
Average operating costs of graphite concentrate (at plant)	US$362.62/T
Average operating costs of graphite concentrate (FOB Fort Dauphin)	US$495.62/T
Equity financing	100%
Inflation	0%
Government gross revenue royalty	2%
Vision Blue gross revenue royalty	3%
Net Smelter Royalty (NSR)	1.5%
Madagascar corporate tax rate	20%

22.3 Financial Model and Results

The Phase 2 revenues, operating and capital costs over a 26 LoM are summarized in Table 64 and Table 66. All values are expressed in millions of US$.

Table 64: Financial Analysis Summary

Item	Amount US$ million
Revenue	**4,847.6**
Mining	(573.0)
Processing	(812.9)
Transportation	(524.0)
G&A and Admin	(39.4)
Total OPEX	**1,949.2**
EBITDA	**2,898.4**
Depreciation	162.1
Government Royalty	97.0
Vision Blue Royalty	145.4
NSR Royalty	43.5
Income Taxes	547.3
Net Profit	**1,903.2**
Initial Capital Expenditures	155.8
Initial Working Capital	20.9
Sustaining Capital Expenditures	15.0
Closure and Rehabilitation Costs	9.5

Table 65: Financial Analysis Results by year

Project Year	Total	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Revenues	4,847.6	-	-	101.2	158.2	207.2	210.4	194.1	183.7	178.3	216.4	185.3	177.0	207.2	188.2	177.7	192.4	190.8	182.3
Mining	(573.0)	-	-	(12.9)	(21.0)	(24.8)	(23.2)	(25.4)	(24.9)	(28.1)	(27.0)	(26.8)	(26.1)	(27.9)	(28.3)	(28.4)	(27.4)	(20.7)	(19.4)
Processing	(812.9)	-	-	(16.1)	(24.7)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)	(32.2)
G&A	(39.4)	-	-	(0.8)	(1.3)	(1.7)	(1.7)	(1.6)	(1.5)	(1.4)	(1.8)	(1.5)	(1.4)	(1.7)	(1.5)	(1.4)	(1.6)	(1.6)	(1.5)
Transportation	(524.0)	-	-	(10.9)	(17.1)	(22.4)	(22.7)	(21.0)	(19.9)	(19.3)	(23.4)	(20.0)	(19.1)	(22.4)	(20.3)	(19.2)	(20.8)	(20.6)	(19.7)
Initial capital costs	(155.9)	(155.9)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital costs	(15.0)	-	-	-	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)
Closure costs	(9.5)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Working capital	-	-	(20.4)	(0.5)	0.6	0.3	(0.1)	0.4	(0.6)	(0.8)	1.0	0.4	(1.3)	0.5	0.3	(0.2)	1.7	0.6	(0.7)
Project Cash Flows (Pre-Tax and Royalties)	**2,718.1**	**(155.9)**	**(20.4)**	**59.9**	**94.0**	**125.8**	**129.9**	**113.7**	**104.0**	**95.9**	**132.4**	**104.6**	**96.3**	**122.9**	**105.6**	**95.8**	**111.6**	**115.8**	**108.2**
Government royalty	(97.0)	-	-	(2.0)	(3.2)	(4.1)	(4.2)	(3.9)	(3.7)	(3.6)	(4.3)	(3.7)	(3.5)	(4.1)	(3.8)	(3.6)	(3.8)	(3.8)	(3.6)
Vision Blue royalty	(145.4)	-	-	(3.0)	(4.7)	(6.2)	(6.3)	(5.8)	(5.5)	(5.3)	(6.5)	(5.6)	(5.3)	(6.2)	(5.6)	(5.3)	(5.8)	(5.7)	(5.5)
NSR royalty	(43.5)	-	-	(0.9)	(1.4)	(1.9)	(2.0)	(1.7)	(1.6)	(1.5)	(2.0)	(1.6)	(1.5)	(1.8)	(1.6)	(1.4)	(1.7)	(1.7)	(1.6)
Taxes	(547.3)	-	-	(10.8)	(17.6)	(24.0)	(24.9)	(21.5)	(19.8)	(18.2)	(25.2)	(19.7)	(18.4)	(23.4)	(19.9)	(18.1)	(20.9)	(21.9)	(20.7)
Project Cash Flows (Post-Tax and Royalties)	**1,884.9**	**(155.9)**	**(20.4)**	**43.1**	**67.2**	**89.6**	**92.5**	**80.8**	**73.4**	**67.3**	**94.5**	**74.1**	**67.5**	**87.3**	**74.7**	**67.4**	**79.5**	**82.6**	**76.8**

The economic analysis of Phase 2 revenues, operating and capital costs over a 26 LoM using discounted cash flow methods is summarized in Table 66 below. The financial model was run at discount rates of 8%, 10% and 12%.

Table 66: Economic Metrics with Varying Discounts

Metric	Values
Before tax and royalties	
Total Project Cash Flows (1)(2)(4)(5)	US$2,718.1 million
NPV @ 8%	US$904.8 million
NPV @ 10%	US$716.7 million
NPV @ 12%	US$574.4 million
IRR	40.4%
Payback Period	3.18 years
After tax and royalties	
Total Project Cash Flows (1)(2)(3)(4)(5)	US$1,884.9 million
NPV @ 8%	US$593.0 million
NPV @ 10%	US$459.3 million
NPV @ 12%	US$358.2 million
IRR	31.4.0%
Payback Period	3.74 years

(1) Assumes Project is financed with 100% equity.

(2) Capex includes process equipment, civil and infrastructure, mining, buildings, electrical infrastructure, project and construction services.

(3) Assumes 2% government gross revenue royalty, 3% Vision Blue gross revenue royalty, 1.5% NSR royalty and corporate tax rate of 20%.

(4) Assumes no inflationary adjustments in sales price, or operating costs.

(5) Based on current market prices provided by UK-based commodity price reporting agencies Benchmark Minerals Intelligence and fast markets.

22.4 Sensitivity Analysis

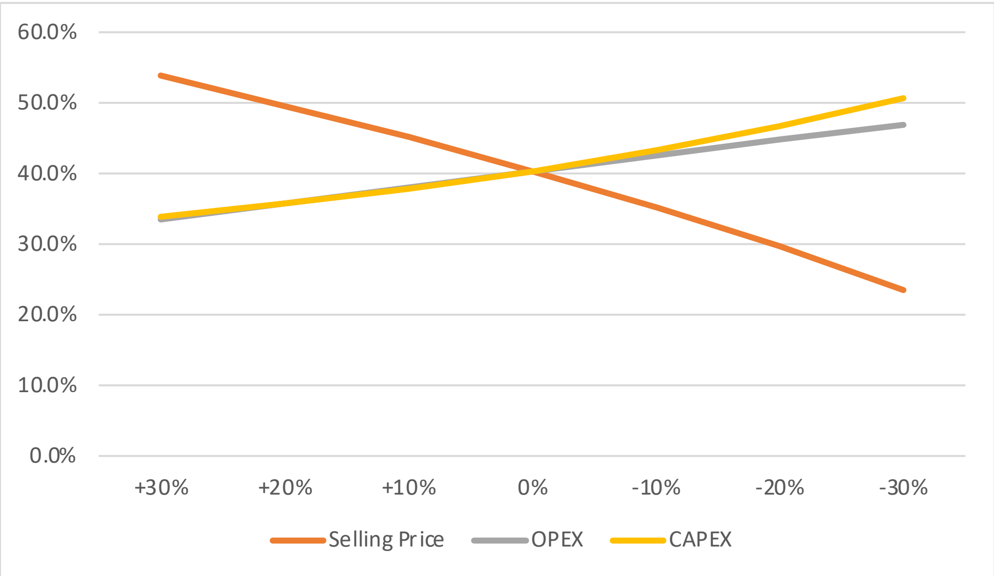
The financial model IRR and NPV and the impact of +/- 10% change to the selling price, operating expenditures and capital expenditures are presented and ranked in order of highest sensitivity to least sensitive in Table 67 below.

Table 67: IRR Sensitivity

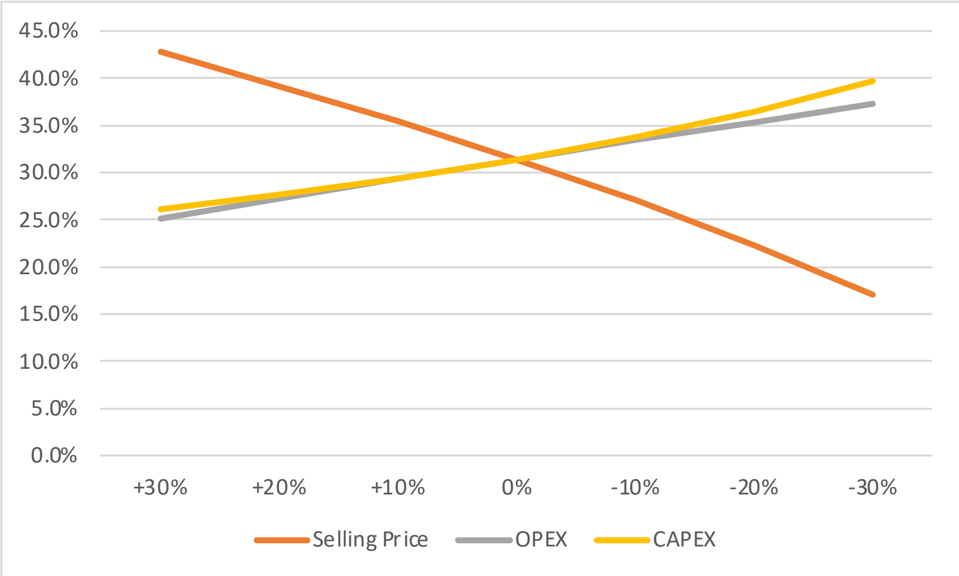
IRR	Pre-Tax and Royalties	Post-Tax and Royalties
No changes (reference point)	*40.4%*	*31.4%*
Selling price +/- 10%	45.1% / 35.2%	35.4% / 27.0%
Operating expenses +/- 10%	38.1% / 42.6%	29.3% / 33.4%
Capital expenditures +/- 10%	37.9% / 43.2%	29.4% / 33.7%
NPV 8%	**Pre-Tax and Royalties**	**Post-Tax and Royalties**
No changes (reference point)	*US$904.8*	*US$593.0*
Selling price +/- 10%	US$1,087.6 / US$722.1	US$727.4 / US$458.7
Operating expenses +/- 10%	US$828.8 /US$980.8	US$533.0 /US$653.0
Capital expenditures +/- 10%	US$889.2 / US$920.4	US$578.7 / US$607.4

22.4.1 Spider Sensitivity Analysis

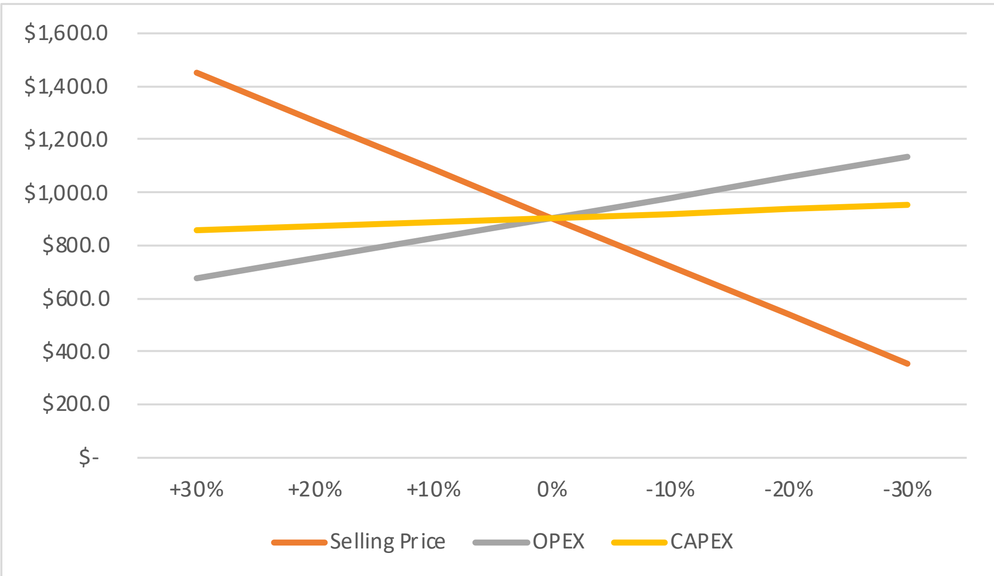
The financial model base IRR and NPV and the sensitivity to +/- 10%, 20% and 30% changes to the selling price, operating expenditures and capital expenditures are presented in graphical form below. The intersection of the lines represents the reference point result from Table 67.



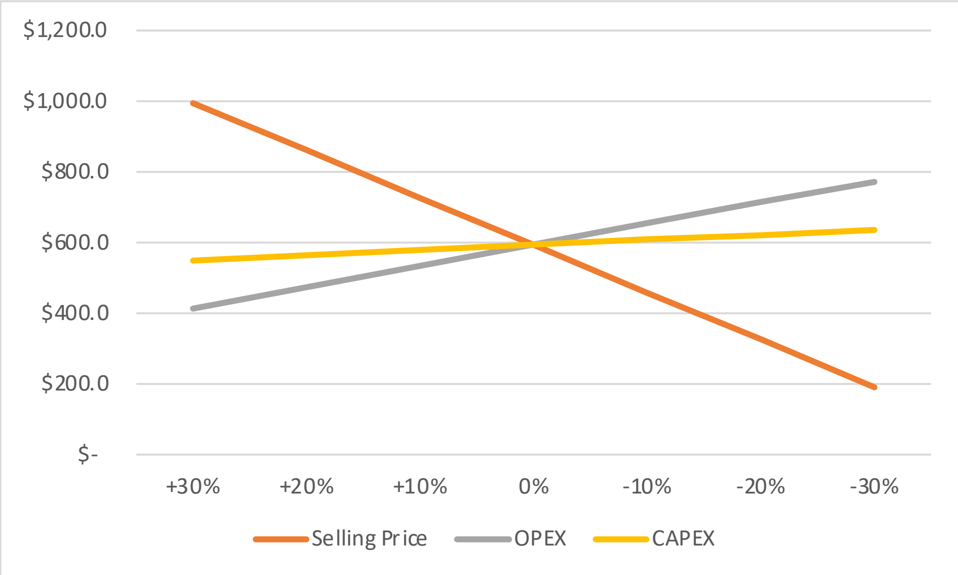
Graph 8: Sensitivity of Project IRR (Pre-tax and royalties) to changes in Selling Price, OPEX and CAPEX



Graph 9: Sensitivity of Project IRR (Post-tax and royalties) to changes in Selling Price, OPEX and CAPEX



Graph 10: Sensitivity of Project NPV (Pre-tax and royalties) to changes in Selling Price, OPEX and CAPEX



Graph 11: Sensitivity of Project NPV (Post-tax and royalties) to changes in Selling Price, OPEX and CAPEX

23 ADJACENT PROPERTIES

No similar graphite deposits are known in south-western Madagascar other than those already detailed in this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any other information on the properties that would affect their interpretations, or conclusions regarding the Molo Graphite Deposit. It should, however, be noted that Molo is only one of nineteen graphite prospects on the greater Property, only seven of which have been drill tested.

24.1 Report on the Hazard (Hazard Study 2) Study carried out on the Proposed Molo Graphite Plant in Madagascar

Two Hazard reports, namely, Hazard on Graphite Dust Exposure and Hazard on Silica Dust Exposure, were completed by Globesight. The reports are based on the Hazard study completed for Phase 1 of the FS, which also applies to Phase 2:

- Crushing.
- Milling and flash flotation.
- Rougher flotation.
- Cleaner circuits.
- Thickening.
- Filtration and drying.
- Bagging.
- Tailings handling.

- Reagent storage and supply.
- Supply of services.
- Possible causes.
- Consequences.
- Preventative avoidance measures.
- Protective mitigating measures.
- Actions needed if the measures were inadequate.
- Determination of causes of the hazard.
- Determination of the consequences of the hazard.
- Evaluation of any existing safeguards / controls (preventative or mitigating).
- New safeguards if existing are judged inadequate.
- Action and responsibilities to execute the action.

25 INTERPRETATIONS AND CONCLUSIONS

25.1 Geology

The Company's 2011 exploration program delineated a number of new graphitic trends in southern Madagascar. The resource delineation drilling undertaken during 2012 to 2014 focused on only one of these, the Molo Deposit, and this has allowed for an Independent, CIM compliant, updated resource statement for the Molo deposit.

The total Measured and Indicated Resource is estimated at 100.37 Mt, grading at 6.27% C. Additionally, an Inferred Resource of 40.91 Mt, grading at 5.78% C is stated. When compared to the November 2012 resource statement (Hancox and Subramani, 2013), this shows a 13.7% increase in tonnage, a 3.4 % decrease in grade, and a 9.8% increase in graphite content. The reason for the increase in tonnage is due to the 2014 drilling on the previously untested north-eastern limb of the deposit, which added additional new resources. Additionally, 23.62 Mt, grading at 6.32% Carbon, have been upgraded by infill drilling from the Indicated to Measured Resource category.

25.2 Mining

Mineral reserves of 22,436,685t (Proven 14,169,741t at 7.00% C and Probable 8,266,944t at 7.04% C) have been declared in the FS with an average grade of 7.02% C. Based on the information contained in Phase 1 of the FS and in this PEA, it is possible to economically mine this deposit.

25.3 Tailings

Tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining and on the same basis as Molo phase 1. In the next phase of the study a detailed design will be completed, complete with environmental and social impact assessment and closure.

25.4 Risks

In addition to the qualitative risk assessment completed during the Molo 2015 FS, a comprehensive HAZARD study was completed as part of the Molo Phase 1 study and applies to this study in equal measure.

25.5 Permitting

While the major permits have been granted for the Project, the Global Environmental Permit and the Mining Permit, various supplementary sectoral permits will need to be obtained.

25.6 Metallurgical Test Work

Comprehensive metallurgical test programs culminated in a process flowsheet that is capable of treating the Molo ore using conventional and established mineral processing techniques. This was done during the Molo Phase 1 study

Process risks associated with the variability with regards to metallurgical performance have been mostly mitigated through the addition of an upgrading circuit. The upgrading circuit treated the combined concentrate after the secondary cleaning circuit. Reduced flake degradation and an improved process flexibility may be obtained by employing separate upgrading circuits for the coarse and fine flakes.

26 RECOMMENDATIONS

26.1 Geology

No further recommendations.

26.2 Mining

Nothing further to report.

26.3 Metallurgical Test Work

Investigate the metallurgical impact of different attrition mill technologies such as stirred media mills, or attrition scrubbers.

Evaluate a range of different grinding media (e.g. different size, shape, material) to determine if flake degradation can be reduced without affecting the concentrate grade.

Develop a grinding energy versus concentrate grade relationship for the best grinding media. This will allow a more accurate prediction of the required attrition mill grinding energy as a function of the final concentrate grade.

Conduct attrition mill vendor tests to aid in the sizing of the equipment.

Carry out Vendor testing on graphite tailings using the optimized reagent regime proposed by the reagent supplier.

Complete a series of flotation tests on samples covering mine life intervals for the 2017 FS pit design.

No further metallurgical test work is required.

26.4 Recovery Methods

The process plant has been designed to easily optimize the final product grade. This is achieved by having two options in the attrition cleaning step.

26.5 Infrastructure

The following are recommended prior to the detailed design stage:

Additional geotechnical investigations at the proposed new construction and permanent camp site, particularly at the location of the new potable water storage tanks.

A detailed geotechnical investigation will need to be undertaken to identify and confirm suitable sources of concrete aggregate and concrete sand materials at the location of the Project site. This testing will need to include for concrete material testing and the production of concrete trial mixes with the material identified.

The geotechnical information will also need to confirm the suitability for construction of all the material to be excavated from the return water dam. It is proposed that all the material excavated from the return water dam is utilized in the works as processed fill material.

Confirmation as to whether the material from the proposed borrow pit near Fotadrevo, (which will be used to supply all fill material for the co-disposal starter wall construction) can be utilized as fill material, or if this material can be stabilized in some manner and used in the works.

A detailed topographical survey will need to be undertaken of the proposed construction site, borrow pit areas and the access road between Fotadrevo and the mine site. This information is required prior to the final detailed design of the plant layout and associated earthworks.

26.6 Water

The following is recommended during the detailed design phase:

Water quality and quantity data is required to provide a baseline for comparison once the Molo Mine is commissioned. To provide the necessary baseline data, regular ground and surface water quality monitoring must be carried out leading up to the date when the Molo Mine will be commissioned. Additionally, proposed production and monitoring boreholes must be installed. This also should include the installation of flow meters on relevant pipelines to verify the dynamic water balance with measured flow rates during operations.

26.7 Environmental, Social

Clean and/or renewable energy supply should be considered as a medium to long term target.

Appointment of a community representative and the establishment of a mandate to sensitize the local communities prior to any Project activities.

Monitoring and auditing to commence at Project preparation phase.

It is recommended that actual activity data, (e.g. kilometers travelled, or liters of diesel consumed) for a financial year is used when a GHG Assessment is being calculated.

Community recruitment, skills development and training should begin at Project preparation phase.

26.8 Permitting

Security of land tenure is a process and is estimated to take 6 to 9 months, thus this process should be commissioned as early as possible prior to Phase 1. The total area concerned is anticipated to be sufficient for the proposed Phase 2 expansion.

Application for all other necessary permits (water use, construction, mineral processing, transportation, export, labour and so forth should be undertaken prior to Phase 2.

Compilation of a comprehensive legal register.

Application for an amendment of the requisite social, resettlement, environmental and construction approvals would be required for the expansion to 2,500,00 ktpa (Phase 2).

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28 CERTIFICATES OF AUTHORS

ERUDITE STRATEGIES | PROJECTS | ENGINEERING

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

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b) Title and Effective Date of Technical Report:

Molo Phase 2 PEA, National Instrument 43-101 Technical Report

Effective date: 27 April 2022.

c) Qualifications:

I graduated with a B Eng (Civil Engineering) degree from the University of Pretoria in 1992.

I am a registered Professional Engineer with ECSA, Registration Number: Pr.Eng 970123

I completed the Director Development Programme through the University of Cranfield (England) in 2013.

My professional memberships include:

University of Pretoria: Member of the Advisory Board of the Engineering Faculty to ensure that the curriculum remains aligned with global standards and practices. I am also a part-time lecturer for final year students in engineering and Project management.

Engineering Council of South Africa (ECSA)

South African Institute of Civil Engineers (SAICE)

d) Site Inspection:

I visited the site on various dates in 2015, 2016 and 2017.

e) Responsibilities:

I was responsible for the following: Sections 1.1 to 1.3, 1.4, 1.8, 1.10,1.11, 1.12, 1.13 to 1.17 to 1.22, Section 2, Section 3, Section 15, Section 16, Section 17, Section 18, Section 19, Section 20, Section 21, Section 22, Section 24, Section 25.2, 25.4 to 25.6, Section 26.2, 26.4 to 26.9, Section 27.2, 27.4 to 27.5.

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I have no prior involvement with this Project.

h) **Compliance with NI 43-101:**

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) **Disclosure:**

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 27 April 2022

Johann de Bruin

Director

Erudite Projects (Pty) Ltd

Summary of recent experience:

Year	Client	Commodity	Type	Description
2019	Two Rivers Platinum	PGM	Project	Development of a crushing and materials handling Project
2019	Baobab Vanadium	Vanadium	Concept	Analysis of Project development strategies
2019	IAMGOLD	Gold	BFS	Product recovery Project
2018	South32	Manganese	BFS	Analysis of multiple product handling solutions
2018	Anglo Platinum	PGM	FS	Materials handling and plant optimisation
2018	Anglo Platinum	PGM	FS	Multiple plant improvement studies
2000 - 2017	Multiple	Various	BFS	Development of multiple Feasibility studies in international jurisdictions

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Oliver Peters

Metpro Management Inc., 102 Milroy Drive, Peterborough, Ontario, K9H 7T2, Canada

Principal Metallurgist

b) Title and Effective Date of Technical Report:

Molo Phase 2 PEA, National Instrument 43-101 Technical Report

Effective date: 27 April 2022.

c) Qualifications:

I graduated with a M.Sc. degree in mineral processing from the RWTH Aachen University.

I am a Registered Professional Engineer with the Professional Engineers of Ontario – Reg. No. 100078050

I have completed various graphite studies as summarized below. I have practiced my profession from 1998 to now.

I have read the definition of "qualified person" set out by National Instrument 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 fulfil the requirements to be a "qualified person" for the purpose of NI 43-101. My relevant experience includes 17 years as a mineral processing engineer and Project manager.

d) Site Inspection:

I did not visit the site

e) Responsibilities:

I was responsible for the following sections: 1.9, 1.18.3, 1.19.3, Section 13, Sections 25.3, 26.3, 27.3

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I was responsible for managing all laboratory and pilot scale programs at SGS Lakefield in the role of a consulting metallurgist.

h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) Disclosure:

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 27 April 2022

Oliver Peters

Principal Metallurgist

Summary of selected recent graphite experience:

Year	Client	Type Of Study	Project Description
2011	Northern Graphite	Laboratory	Process development
2011	Northern Graphite	Pilot Plant	Validation of flowsheet, collection of engineering data
2012/2013	Focus Graphite	Laboratory	Process development, variability study
2012	Focus Graphite	Pilot Plant	Validation of Flowsheet, collection of engineering data
2012	Magnesita	Laboratory	Process development, variability study
2012	Mason Graphite	Laboratory	Process development
2013	Canada Carbon	Laboratory	Process development
2013	Graphite One	Laboratory	Process development
2014	Alabama Graphite	Laboratory	Process development
2014	Sovereign Metals	Laboratory	Process development
2014	Nouveau Monde	Laboratory	Process development
2014	Dalgraphite	Laboratory	Process development
2014	Canada Carbon	Pilot Plant	Validation of Flowsheet, collection of engineering data
2013-2015	NextSource Materials	Laboratory	Process development, variability study
2014	NextSource Materials	Pilot Plant	Validation of Flowsheet, collection of engineering data

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Sivanesan (Desmond) Subramani

30 7th Avenue, Parktown North, Randburg, Gauteng, South Africa

Principal Resource Geologist

b) Title and Effective Date of Technical Report:

Molo Phase 2 PEA, National Instrument 43-101 Technical Report

Effective date: 27 April 2022.

c) Qualifications:

I graduated with a B.Sc. Honours (Geology and Economic Geology) degree from the University of KwaZulu Natal - Durban in 1994.

I am a member in good standing of the South African Council for Natural Scientific Professions (SACNASP No. 400184/06) as well as a Member of the Geological Society of South Africa.

I have practiced my profession from 1995.

I have read the definition of “qualified person” set out by National Instrument 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 fulfil the requirements to be a “qualified person” for the purpose of NI 43-101. My relevant experience includes 19 years as a Geologist, of which the last 8 years focused on geostatistical resource modelling and estimation.

d) Site Inspection:

I visited the site 15 to 19 February 2014.

e) Responsibilities:

I was responsible for the following sections: Section 1.6, Section 12, Section 14.

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I was responsible for the sections 12 and 14 of the previous National Instrument 43-101, filed on 12 April 2013

h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) Disclosure:

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 27 April 2022

Desmond Subramani

Principal Resource Geologist

Summary of recent experience:

Year	Client	Commodity	Type Of Study	Project Description
2005 - 2006	Katanga Mining	Copper & Cobalt	PFS	Generated Cu and Co Mineral Resources for a Prefeasibility study used in the listing on the Toronto Stock Exchange - DRC
2007	Anglo Platinum	Platinum Group Metals	PFS	Completed PGE Mineral Resource update for Ga Pasha Project prefeasibility study – South Africa
2007 - 2008	Lonmin Platinum	Platinum Group Metals	Internal Report	Provided technical advice and supervision in generating mineral resources for Baobab Platinum mine. – South Africa
2011	INV Metals	Copper	Resource Estimate	Mineral Resource Estimations, plus Ni43-101 for Okohongo Cu prospect - Namibia
2011	Nkwe Platinum	Platinum Group Metals	FS	Mineral Resource Estimations for their Garatau PGE Project, as part of a Feasibility Study. – South Africa
2011-2014	AngloGold Ashanti	Gold	Annual Resource Report	Annual update and reconciliation of resources for Siguiri Gold Mine. – Guinea Basao.
2011-2012	Metorex Group	Copper	PFS	Update Mineral Resources for Kinsenda Copper Project, as part of an ongoing feasibility study. - DRC
2012/2014	NextSource Materials	Graphite	PFS	Mineral Resource Estimations for their Molo graphite Project, as part of a Feasibility Study - Madagascar
2013/2014	Helio Resources	Gold	PFS	Mineral Resource Estimation, as part of the Ni43-101 for their SMP deposit. - Tanzania.

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Philip John Hancox

30 Seventh Avenue, Parktown North, Randburg, Gauteng, South Africa

General Manager

b) Title and Effective Date of Technical Report:

Molo Phase 2 PEA, National Instrument 43-101 Technical Report

Effective date: 27 April 2022

c) Qualifications:

I graduated with a B.Sc. Honours (1990) and Ph.D. (1998) from the University of Witwatersrand - Johannesburg.

I am a member in good standing of the South African Council for Natural Scientific Professions (SACNASP No. No. 400224/04), as well as a Fellow of the Geological Society of South Africa.

I have practiced my profession since 1998.

I have read the definition of “qualified person” set out by National Instrument 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 fulfil the requirements to be a “qualified person” for the purpose of NI 43-101. My relevant experience includes 17 years as a Geologist, of which the last 11 years focused on exploration.

d) Site Inspection:

I visited the site during May of 2012 and 2013.

e) Responsibilities:

I am responsible for the following sections: Sections 1.5, 1.7, 1.18.1, 1.19.1, Section 4, Section 5, Section 6, Section 7, Section 8, Section 9, Section 10, Section 11, Sections 25.1, 26.1, 27.1

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I was responsible for part of Item 1, Items 2 to 13 and items 23 to 27 of the previous National Instrument 43-101, filed on 12 April 2013.

h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) Disclosure:

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 27 April 2022

Philip John Hancox (Pr.Sci.Nat.)

General Manager Africa and Australasia

Caracle Creek International Consulting (Pty) Limited

ERUDITE STRATEGIES | PROJECTS | ENGINEERING

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Pono Mogoera, Process Engineer

24 Adele Street, Kempton Park, 1619

b) Title and Effective Date of Technical Report:

Molo Phase 2 PEA , National Instrument 43-101 Report

Effective date: 27 April 2022

c) Qualifications:

Pono is a registered Professional Engineer (Pr.Eng.) with ECSA with a Bachelor of Science degree in Chemical Engineering with an additional post graduate certificate in Management Advancement Program (Wits Business School).

These qualifications are complemented by over 14 years of working experience in the process engineering field across various sectors. Pono has an array of experience that ranges from the operations of mineral processing to working extensively in the Project and consulting space.

Pono has previously work on two other graphite Project in the Southern African space.

d) Site Inspection:

I have not visited the site.

e) Responsibilities:

I am responsible for the following sections: Section 1.10, Section 17.1, section 18.2, Section 21.2

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

This is the first time I have been involved with this Project.

h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) Disclosure:

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 27 April 2022

Pono Mogoera (Pr Eng)

Process Engineer

Erudite Project (PTY) LTD

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) **Name, Address, Occupation:**

Schalk Pienaar (Pr.Eng)

Civil and Structural Engineer

9 George Storrar Drive, Groenkloof, Pretoria

b) **Title and Effective Date of Technical Report:**

Molo Phase 2 PEA National Instrument 43-101 Technical Report

Effective date: 27 April 2022

c) **Qualifications:**

Etienne is a registered Professional Engineer (Pr.Eng.) with a Bachelor of Science degree in Civil Engineering. Etienne also holds a master's degree in engineering management.

I am a Professional Engineer registered with Engineering Council of South Africa (ECSA) – 20080017.

These qualifications are complemented by over 19 years of working experience across various sectors in the civil and structural engineering industry. Etienne has an array of experience that range from the construction management to the various stages of design development of complex industrial Projects. Etienne has worked on mining, petrochemical, and infrastructure development Projects across the African continent.

d) **Site Inspection:**

I have not inspected the site

e) **Responsibilities:**

I am responsible for the following sections:

18.2 Plant Infrastructure

18.7 Shared Infrastructure and Services

18.7.1 Access Roads

18.7.2 Terraces and Bulk Earthworks

18.7.3 Gate House and Turnstile Access Control

18.7.4 Training Centre

18.7.5 Re-fuelling Station

18.7.6 Waste Water Treatment Plant

18.7.7 Portable Water Treatment Plant

18.7.8 Storm Water Drainage

18.7.9 Storm Water Dams

26.5 Infrastructure

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I was involved with the previous feasibility study for this Project.

h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) Disclosure:

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 27 April 2022

Schalk Pienaar (Pr Eng)

Civil and Structural Engineer

Erudite Projects (Pty) Ltd

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

j) Name, Address, Occupation:

Hercules Albertus Smith

Senior Electrical Engineer

9 George Storrar Drive, Groenkloof, Pretoria

k) Title and Effective Date of Technical Report:

Molo Phase 2 PEA, National Instrument 43-101 Technical Report

Effective date: 30 April 2022

l) Qualifications:

B.Eng Electrical Degree – University of Pretoria 1996

B.Eng Electrical Degree (Hons) – University of Pretoria 2001

Professor in Engineering (200000338) – Engineering Council of SA 2000

m) Site Inspection:

I have not inspected the site

n) Responsibilities:

I am responsible for the following sections:

1. 2.5 Source of information
2. 18.8 Power Generation Facility
3. 18.8.1 Generator Loading
4. 18.8.2 Control and Instrumentation
5. 20.3 Environmental and Social Sensitivities
6. 20.3.3 Electrical supply

o) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

p) Prior Involvement:

Intermittently involved since 2017.

q) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

r) Disclosure:

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 27 April 2022

H.A. Smith (Pr Eng)

Senior Electrical Engineer

Erudite Projects (Pty) Ltd

ERUDITE STRATEGIES | PROJECTS | ENGINEERING

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Albertus Wynand Christoffel (Alkie) Marais

Principal Hydrogeologist

56 Jerome Road, Lynnwood Glenn, Pretoria 0081

b) Title and Effective Date of Technical Report:

Molo Phase 2 PEA, National Instrument 43-101 Technical Report

Effective date: 27 April 2022

c) Qualifications:

Albertus is specialist in ground water services and has 23 years of experience and specialises in Mining related Hydrogeological investigations, ground water contaminate studies, Mine dewatering studies and design, flow and contaminant transport modelling. Hydro-geochemical waste studies, environmental management studies, ground water monitoring programmes and hydro-chemical analysis, ground water resource, aquifer assessment and water balance studies.

I am a registered scientist with South African Council of Natural Scientific Professionals (SACNASP), Registration number 400012/06.

MSc Geohydrology IGS (UFS) Bloemfontein SA 1997.

BSc (Hons) Geology (UFS) Bloemfontein SA 1993.

BSC Geology, Geochemistry (UFS) Bloemfontein SA 1992

d) Site Inspection:

March 2014.

e) Responsibilities:

I am responsible for the following sections:

1. 17.3 Water
2. 17.3.4 Conceptual Storm Water Management Plan
3. 1.20 Environmental and Permitting
4. 1.20.1Environmental and Social Impact Assessment
5. 1.22 Recommendations
6. 1.22.7 Environmental, Social
7. 1.22.8 Permitting
8. 18.5 Raw Water Supply
9. 20 Environmental Studies, Permitting and Social Impact
10. 20.8 Water and Water Management
11. 20.8.1 Mine Waste Management

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

Involved with original Energizer Resources Study for Molo Project 2014-2015.

h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) Disclosure:

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 27 April 2022

A.W.C (Alkie) Marais Principal Hydrogeologist

GCS Water and Environmental Consultants (Pty) Ltd