Konnex Resources Inc.

Empire Pre-Feasibility Study

PFS Report

Project No. 134004

Prepared by:

FORTE DYNAMICS, INC
120 Commerce Drive
Units 3 & 4
Fort Collins, CO 80524
(720) 642-9328

Table of Contents

1. Summary 15

1.1 Introduction 15
1.2 Location 15
1.3 Property Description 16
1.4 Geology and Mineralization 16
1.5 Mineral Resource Estimate 17
1.6 Mining Methods 19
1.7 Metallurgy 19
1.8 Mineral Processing 21
1.9 Project Infrastructure 21
1.10 Market Studies and Contracts 21
1.10.1 Markets 21
1.10.2 Contracts 21
1.11 Social and Environmental Aspects 22
1.12 Capital and Operation Cost Estimates 22
1.12.1 Capital Cost Estimates 22
1.12.2 Operating Cost Estimates 23
1.12.3 Financial Analysis 24
1.13 Conclusions and Recommendations 24
1.13.1 Feasibility Study 24
1.13.2 Geology, Resources, and Mine Planning 24
1.13.3 Mineral Processing 24
1.13.4 Infrastructure 25
1.13.5 Environmental and Social Impacts 25
1.13.6 Results of the Site-Wide Water Balance Model 25
1.13.7 Groundwater Hydrology and Mine Dewatering 25
1.13.8 Process Plant Geotechnical Investigation 25
1.13.9 TSF Design 26

2. Introduction 27

2.1 Background Information 27
2.2 Terms of Reference and Purpose of the Report 28
2.3 Sources of Information 28
2.4 Units of Measure 29
2.5 Detailed Personal Inspections 30
2.5.1 Open-Pit Surface Mine 30
2.5.2 Dry Stacked Tailings 30
2.5.3 Waste Rock Storage Facility 30
2.5.4 Measures to Isolate or Control Toxic or Deleterious Materials 31
2.5.5 Plans for Monitoring Site Conditions During Periods of Non-Operation or Curtailment 31
2.5.6 Procedure for Anticipated Temporary Closures 31
2.5.7 Procedures to Resume Mining Operations 32
2.5.8 Past Inspections 32

3. Reliance on other Experts 33

4. Property Description and Location 34

4.1 Location 34

4.2 Mineral Tenure, Agreements, and Encumbrances 36
4.3 Risks 40

5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography 41

5.1 Access and Climate 41
5.2 Local Resources and Infrastructure 41
5.3 Physiography 41

6. History 42

6.1 History of Previous Exploration 42
6.2 Historical Estimates 45
6.3 Historic Production 45

7. Geological Setting and Mineralization 46

7.1 Regional Geology 46
7.2 Local and Property Geology 51
7.2.1 Lithology 52
7.2.2 Structure 54
7.3 Alteration and Mineralization 56
7.3.1 Skarn Mineralization 56
7.3.2 Iron Oxide Breccia Mineralization 56

8. Deposit Types 58

9. Exploration 60

10. Drilling 61

10.1 Type and Extent 61
10.2 Procedures 63
10.3 Summary and Interpretation of Results 64
10.3.1 Infill Drilling and Zone Confirmation 64
10.3.2 Westward Expansion 64
10.3.3 Inferred Areas and Resource Evaluation 64
10.3.4 Peripheral Exploration and Continued Exploration 64
10.3.5 Metallurgical Testing and Geotechnical Data Collection 65
10.4 Historical Drilling 65

11. Sample Preparation, Analyses, and Security 67

11.1 Sample Preparation 67
11.1.1 ALS Methods 67
11.1.2 Sample Preparation 68
11.2 Quality Control/Quality Assurance 68
11.2.1 Blanks 68
11.2.2 Standard Analysis 71
11.2.3 Duplicate Analysis 74
11.3 Opinion on Adequacy 76

12. Data Verification 77

12.1 Data Verification Procedures 77
12.1.1 Current Personal Inspection 77
12.1.2 Review of Historical Data and Internal Reports 78

12.1.3 Musgrove (2011) 80
12.1.4 HRC Exploration Database Validation 82
12.2 Opinion on Adequacy 82

13. Mineral Processing and Metallurgical Testing 83

13.1 Historical Studies 83
13.1.1 Cambior (1997)[1-4] 83
13.1.2 KCA (2005)[5] 83
13.1.3 KCA (2013)[6] 84
13.1.4 Mineral Technologies Inc (2018)[7] 84
13.1.5 AuRIC (2020)[8] 85
13.1.6 Forte Analytical (2022)[9] 85
13.1.7 CSA Global (2022)[10] 85
13.1.8 Ausenco (2022)[11] 85
13.1.9 Conclusions 86
13.2 Forte Analytical Recent Studies – Phase I (2022-2023)[12] 86
13.2.1 Head Analyses 87
13.2.2 Comminution Tests 88
13.2.3 Gravity Testing 89
13.2.4 Ammonium Thiosulfate Leaching 91
13.2.5 Sulfuric Acid Leaching 94
13.2.6 Flotation Testing 96
13.2.7 Smelter Penalty Analyses of Flotation Concentrates 103
13.2.8 Thickening Test 105
13.2.9 Filtration Tests 105
13.2.10 Cementation Testing 106
13.2.11 Evaluation of Alternative Processing Flowsheets 106
13.2.12 Conclusions 110
13.3 Forte Analytical – Phase II Testing (2024) 110
13.3.1 Preparation of Composites Representing Ores Processed Yearly in the Plant 111
13.3.2 Rougher Flotation Tests 111
13.3.3 Open-Cycle Rougher-Cleaner Flotation Test 112
13.3.4 Acid Leach of Flotation Tailing 115
13.3.5 Carbonate Flotation and Acid Leach 115
13.3.6 Conclusions 116
13.3.7 Recommendations 116

14. Mineral Resource Estimates 118

14.1 Data Sources and Compilation 118
14.2 Implicit Modeling Method 119
14.2.1 Fault Blocks 120
14.2.2 Estimation Domains 122
14.3 Depletion 123
14.4 Compositing 123
14.5 Capping 124
14.6 Variography 124
14.7 Estimation Methodology 125
14.8 Density 127
14.9 Validation 128
14.9.1 Comparison with Nearest Neighbor Models 128
14.9.2 Swath Plots 129
14.9.3 Section Inspection 131

14.10 Mineral Resource Classification 134
14.10.1 Mineral Resource Tabulation 134
14.11 Mineral Resource Statement 135
14.11.1 Resource Sensitivity 136

15. Mineral Reserves 137

15.1 Introduction 137
15.2 Estimation Parameters 137
15.2.1 Pit Slopes 137
15.2.2 Dilution and Losses 137
15.2.3 Pit Optimization and NSR Calculation 138
15.3 Mineral Reserve Statement 139
15.4 Factors that May Affect the Mineral Reserve 140

16. Mining Methods 141

16.1 Overview 141
16.2 Geotechnical 141
16.3 Hydrogeological Considerations 144
16.4 Pit Designs 144
16.5 Waste Rock Storage Facility Designs 148
16.6 Dilution and Mining Losses 149
16.7 Pre-Production Development 149
16.8 Mine Production Schedule 150
16.8.1 Mining Through Underground Workings 152
16.8.2 Production Schedule Parameters 152
16.8.3 Drilling and Blast Parameters 153
16.8.4 Load and Haul Parameters 154
16.9 Mining Equipment 155

17. Recovery Methods 157

17.1 Process Design Criteria 157
17.2 Flow Sheet Development & Description 162
17.3 Description of Process Areas 164
17.3.1 Crushing and Screening 164
17.3.2 Milling (Ball Mill Circuit) 164
17.3.3 Flotation Circuit 164
17.3.4 Acid Leach Circuit, Disc Filters, and Pug Mill 165
17.3.5 Copper Cementation Circuit 165
17.4 Process Water 165

18. Project Infrastructure 167

18.1 Site Layout 167
18.2 Area 2000 – Mine 168
18.2.1 Area 2300 – Mine Support Facilities 168
18.2.2 Area 2400 – Mine Support Services 171
18.3 Area 3500 – Cementation and Tails 171
18.3.1 Area 3540 – Tailings Storage Facility 171
18.4 Area 4000 – Project Services 171
18.4.1 Area 4100 - Power Supply 172
18.4.2 Area 4200 – Communications 173

18.4.3 Area 4300 – Ancillary Facilities 173
18.4.4 Area 4400 – Access Road 174
18.4.5 Area 4500 – Water Management 174
18.4.6 Area 4600 – Security 174
18.4.7 Area 4700 – Site Utilities 175
18.4.8 Area 4800 – Waste Disposal 176
18.4.9 Area 4900 - Plant Mobile Equipment 176
18.5 Area 5000 – Project Infrastructure Development 177
18.5.1 Area 5100 – Site Preparation 177
18.5.2 Area 5600 – Bulk Transport 177

19. Market Studies and Contracts 178

19.1 Markets 178
19.2 Contracts 178

20. Environmental Studies, Permitting, and Social or Community Impact 179

20.1 Environmental Studies 179
20.1.1 DEQ Preliminary Assessment 179
20.1.2 Environmental Baseline Studies 179
20.2 Permitting 180
20.2.1 Idaho Joint Review Process 181
20.2.2 State of Idaho Permits 182
20.2.3 Federal Environmental Review 182
20.3 Geochemical Characterization 183
20.3.1 Potential Acid Generation 183
20.3.2 Leaching Tests 184
20.3.3 Waste Rock 184
20.3.4 Tailings 184
20.4 Water Management 186
20.5 Social and Community Impact 186
20.5.1 Cultural, Social, and Economic Resources 186
20.5.2 Employment 187
20.5.3 Taxes and Royalties 187
20.5.4 Community Impacts 187
20.5.5 Community Engagement and Relations 187
20.5.6 Tribal Consultation 188
20.6 Closure and Reclamation Planning 188
20.6.1 Reclamation Standards and Approach 188
20.6.2 Reclamation and Financial Assurance Cost Estimates 189

21. Capital and Operating Costs 191

21.1 Capital Costs Summary 191
21.2 Capital Cost Estimating Methodology for Mining 192
21.3 Mine Capital Costs 192
21.4 Capital Cost Estimating Methodology for Process and Infrastructure 194
21.4.1 Mechanical Equipment Total Installed Capital Cost Estimating Methodology 194
21.4.2 Pre-Configured Mechanical Equipment Total Installed Capital Cost Estimating Methodology 195
21.4.3 Freight Costs 195
21.4.4 Other Areas Capital Cost Estimate 195
21.4.5 Quantity Development and Unit Rates 197
21.4.6 Area 8000 - Management, Engineering, EPCM Services 198

21.4.7 9000 Preproduction Costs 198
21.4.8 10000 Asset Sale 199
21.5 Process Plant & Infrastructure Capital Costs 200
21.5.1 3000 Process Plant 200
21.5.2 4000 Project Services & Infrastructure – General 200
21.5.3 8000 Management, Engineering, EPCM Services 200
21.5.4 9000 Pre-Production Process Plant Costs 201
21.6 Operating Costs 201
21.6.1 Mine Operating Costs 202
21.6.2 Process and Infrastructure Operating Costs 204
21.6.3 Process Plant Inputs 207
21.6.4 General and Administrative Operating Costs 208

22. Economic Analysis 210

22.1 Summary 210
22.2 Prices and Revenue 211
22.3 Capital Costs 212
22.4 Operating Costs 212
22.5 Taxes 215
22.6 Royalties 215
22.7 Financial Model 215
22.8 Sensitivities 221
22.8.1 Copper Price 221
22.8.2 Operating Costs, Capital Costs and Metal Price Sensitivity 222

23. Other Relevant Data and Information 223

23.1 Process Plant Geotechnical 223
23.2 Water Management 223
23.2.1 Water Balance 223
23.2.2 Water Management Infrastructure 223
23.3 Surface Water Hydrology 224
23.3.1 Water Balance 225
23.4 Project Implementation 225
23.4.1 Project Implementation Strategy 225
23.4.2 Project Organization 225
23.4.3 EPCM Management 226
23.4.4 Engineering 226
23.4.5 EPCM Controls 226
23.4.6 Procurement 227
23.4.7 Construction Management 228
23.4.8 Commissioning 228
23.4.9 Temporary Construction Facilities 229
23.4.10 Industrial Relations 229
23.4.11 Health and Safety 229
23.4.12 Environment 230
23.4.13 Schedule 230

24. Interpretation and Conclusions 234

24.1 Project Risks 234
24.2 Geology and Resources 234
24.3 Mineral Reserve and Mine Planning 234

24.4 Mineral Processing 234
24.4.1 Site Preparation 234
24.4.2 Support Buildings 234
24.4.3 Access Roads 235
24.4.4 Heavy Lifts 235
24.4.5 Bulk Transport 235
24.4.6 Communications 235
24.5 Project Services 235
24.6 Environmental and Social Conclusions 235
24.7 Recommendations Resource and Exploration 236
24.8 Mining Risks and Opportunities 236
24.9 Mineral Processing 236
24.10 Environmental Studies 236
24.11 Site-Wide Water Balance 236
24.12 Groundwater Hydrology and Mine Dewatering 236
24.13 Environmental and Social Conclusions 236
24.14 Tailings Facility Design 236
24.14.1 Followed by laboratory testing and reporting.TSF Construction Schedule 237
24.14.2 Geotechnical Investigation and Assessment 237
24.14.3 Waste Rock Testing 237
24.14.4 Consolidation/Seepage Modeling 237
24.14.5 Water Balance 237
24.14.6 TSF Consequence Classification 237
24.15 Process Operating Costs 238

25. References 239

Figures

Figure 1-1: Empire Mine Project Location 15
Figure 1-2: Overall Process Flowsheet 20
Figure 4-1: Empire Mine Project Location 34
Figure 4-2: Empire Mine Project Claim Areas 39
Figure 4-3: Empire Project Honolulu and Mackay Group Claim Areas 40
Figure 7-1: A Regional Geologic Map Portraying the Idaho-Montana Fold-Thrust Belt, extending from Central Idaho to Southwestern Montana (Not Shown). 47
Figure 7-2: Map of the North American Cordillera, illustrating its Major Tectonic Features 49
Figure 7-3: Map Adapted from Sims et al. (2005), Focusing on the Distribution of Basement Structures in Idaho and Highlighting Major Intrusive Events 50
Figure 7-4: Map of the White Knob Horst and Empire Mine (Farwell & Full, 1944; Skipp et al., 2009) 51
Figure 7-5: Map of the Empire Mine (Farwell & Full, 1944) 54
Figure 7-6: Schematic Illustration of Various Arc Stress States and Tectonomagmatic Settings of Convergent Margins (Rhys et al., 2020). 55
Figure 8-1: Schematic illustration of the Geological Setting and Characteristics of Low- and High-Sulfidation Epithermal Au-Cu Deposits and Sub-Volcanic Porphyry-Type Cu-Au Deposits. 59
Figure 9-1: Copper Concentration in Channel Samples, Displayed as a Heat Map 60
Figure 10-1: Konnex Drillhole Collar Locations 62
Figure 10-2: Historic Drillhole Collar Locations 66
Figure 11-1: Copper Blank Analysis 69
Figure 11-2: Silver Blank Analysis 70
Figure 11-3: Gold Blank Analysis 70
Figure 11-4: Z-Score Plot of Copper Standards 72
Figure 11-5: Z-Score Plot of Silver Standards 73
Figure 11-6: Z-Score Plot of Gold Standards 73
Figure 11-7: Absolute Relative Differences Plot for Copper Duplicates 74
Figure 11-8: Absolute Relative Differences Plot for Silver Duplicates 75
Figure 11-9: Absolute Relative Differences Plot for Gold Duplicates 75
Figure 13-1: Flowsheet for Gravity/Flotation Testing of Master Composites (Tests 1-8) 89
Figure 13-2: Gravity Testing Flowsheet for High Grade Magnetite Master Composites (Tests 9-11) 90
Figure 13-3: Rougher and Cleaner Open-Circuit Test Flowsheet 98
Figure 13-4: Locked-Cycle Test Flowsheet 99
Figure 13-5: Copper Recovery as Function of Feed Grade 102
Figure 13-6: Gold Recovery as Function of Feed Grade 103
Figure 13-7: Simplified Gravity/Flotation Flowsheet 107
Figure 13-8: Simplified ATS/Acid Leach Flowsheets 108
Figure 13-9: Simplified Flotation/Acid Leach Process Flowsheet 109
Figure 13-10: Acid Consumption vs Carbonate in Ore 116
Figure 14-1: Ternary Classification Diagram for Rock Unit Classification Based on Ca, Fe, and Al/Ti ratio. 119
Figure 14-2: Oblique View of the General Workflow to Construct the Geologic Model 121
Figure 14-3: East-West Copper Swath Plot 130
Figure 14-4: North-South Copper Swath Plot 130
Figure 14-5: Elevation Copper Swath Plot 131
Figure 14-6: Bench 8530 of Estimated Copper Grades with Composites and Section Location Lines 132
Figure 14-7: N807,510 Cross Section of Estimated Copper Grades with Composites 133

Figure 14-8: N807,820 Cross Section of Estimated Copper Grades with Composites 133
Figure 14-9: N807,960 Cross Section of Estimated Copper Grades with Composites 134
Figure 14-10: Pit Optimization Copper Sensitivity Chart 136
Figure 15-1: Pit Optimization Results 139
Figure 16-1: Recommend Slope Angles by Sector 143
Figure 16-2: Phase 1 Pit Design 145
Figure 16-3: Phase 2 Pit Design 146
Figure 16-4: Phase 3 Pit Design 147
Figure 16-5: Waste Rock Storage Facility and Low Grade Ore Storage Facility 148
Figure 16-6: Pre-Production Haul Roads 149
Figure 16-7: Yearly Ore Schedule 151
Figure 16-8: Yearly Waste Schedule 151
Figure 17-1: Overall Process Flowsheet 163
Figure 18-1: Empire Mine Project Location 167
Figure 18-2: Site Layout Map 168
Figure 18-3: TSF and WRSF Layout 169
Figure 18-4: Truck Maintenance Pad General Arrangement 170
Figure 18-5: Utility Layout 172
Figure 22-1: Mine Operating Cost Distribution 213
Figure 22-2: Process Operating Cost Distribution 214
Figure 22-3: G&A Operating Cost Distribution 215
Figure 22-4: Project Copper Price Sensitivity Analysis 221
Figure 22-5: Project Operating Cost, Capital Cost and Metal Price Sensitivity Analysis 222
Figure 23-1: Konnex Project Organization Chart 226
Figure 23-2: Project EPCM Schedule 233

Tables

Table 1-1: Parameters Used for Resource Pit Shell Generation 18
Table 1-2: Mineral Resource Statement for the Empire Mine, Custer County, Idaho, U.S.A., Hard Rock Consulting, LLC, January 1, 2024 19
Table 1-3: Estimated Capital Cost Summary (USD $000s) 22
Table 1-4: Mining Operating Costs (USD) 23
Table 1-5: Estimated Processing Plant Operating Costs (USD) 23
Table 1-6: Total Cash Operating Costs (USD) 23
Table 1-7: Summary of Economic Results (USD) 24
Table 4-1: Empire Mine Project Mining Claims 36
Table 6-1: Historic Operators of the Empire Mine (Mitchell, 1997) (Reproduced) 43
Table 10-1: Summary of Konnex Drillhole Totals by Year, Area, and Type 61
Table 10-2: Summary of Drillhole Totals by Year, Operator, and Drillhole Type for Historic Drilling 65
Table 11-1: Suite of Elements Analyzed in ALS ME-ICP61 67
Table 11-2: MEG-AG-2 Standard Tolerance and Average Grade Information 71
Table 11-3: MEG-AG-3 Standard Tolerance and Average Grade Information 71
Table 11-4: MEG-CU-1 Standard Tolerance and Average Grade Information 71
Table 11-5: A106010X Standard Tolerance and Average Grade Information 71
Table 11-6: A106013X Standard Tolerance and Average Grade Information 72
Table 13-1: Head Analyses of Composite Samples 87
Table 13-2: Whole Rock Analyses of Endo/Exo and Magnetite Master Composites 87
Table 13-3: Comminution Test Results 88
Table 13-4: Indirect Bond's Ball Mill Work Indices 88
Table 13-5: Gravity Test Results for Endo/Exo and Magnetite Master Composites 89
Table 13-6: Gravity Test Results for High Grade Magnetite Master Composite 90
Table 13-7: Ammonium Thio-Sulfate Leach (ATS) at Varying Grind Sizes for Endo/Exo and Magnetite Master Composites 92
Table 13-8: Effect of ATS Concentration on Metal Extraction at P_{80} of 65 mesh for Endo/Exo and Magnetite Master Composites 92
Table 13-9: Effect of Recycling ATS on Metal Extraction at P_{80} of 100 Mesh for Endo/Exo and Magnetite Composite 93
Table 13-10: ATS Testing of Variability in Flotation Tailing Samples 93
Table 13-11: Summary of Acid Leach Test on Flotation Tailings at Varying Grind Sizes 95
Table 13-12: Summary of Acid Leach Test on Flotation Tailings for Variability Samples 95
Table 13-13: Summary of Flotation Test Results for Endo/Exo and Magnetite Master Composites for Varying Grind Sizes 97
Table 13-14: Summary of Flotation Test Results for Endo/Exo and Magnetite Master Composites for Varying Grind Sizes and Sulfidization 97
Table 13-15: Rougher Flotation Test at P_{80} of 100 Mesh in One Cubic Foot Flotation Cell for Endo/Exo Master Composite (Test No. 17) 98
Table 13-16: Cleaner Flotation Tests With/Without Regrind of Rougher Concentrate from Flotation Test No. 17 99
Table 13-17: Summary of Locked-Cycle Test Results 100
Table 13-18: Flotation Test Results for Variability Samples 101
Table 13-19: Cleaner Concentrate Grades for Variability Samples 101
Table 13-20: Smelter Penalty Analyses of Copper Concentrates 103
Table 13-21: Chemical Penalty Analyses of Copper Concentrates 104

Table 13-22: Thickening Test Results 105
Table 13-23: Summary of Vacuum and Pressure Filtration Results 105
Table 13-24: Solution Analyses of Pre- and Post-Cementation for Endo/Exo Leach Test 106
Table 13-25: Analyses of Cemented Copper 106
Table 13-26: Gravity and Flotation Results for the Two Master Composites at P_{80} = 100 mesh 107
Table 13-27: Combined Results for Two-Stage Leach Process at P_{80} = 100 mesh 108
Table 13-28: Metal Recoveries for Flotation/Acid Leach Process for Endo/Exo Master Composite 110
Table 13-29: Yearly Composite Sample Head Analyses 111
Table 13-30: Rougher Flotation Results for Varying Grind Sizes 111
Table 13-31: Flotation Mass Balance for Year 1 Composite Sample 113
Table 13-32: Flotation Mass Balance for Year 2 Composite Sample 113
Table 13-33: Flotation Mass Balance for Year 3 Composite Sample 114
Table 13-34: Flotation Mass Balance for Year 4-9 Composite Sample 114
Table 13-35: Acid Leach Kinetic Data for Flotation Tailing 115
Table 13-36: Comparison of Leaching of Sulfide and Carbonate Flotation Tailing 117
Table 14-1: Model Lithology Codes 120
Table 14-2: Geologic Model Comparison to 20-foot Composites 122
Table 14-3: Domain Composite Data 123
Table 14-4: Capping Strategy 124
Table 14-5: Indicator Domain Variogram Parameters 125
Table 14-6: Outside and OVB Domain Variogram Parameters 125
Table 14-7: Estimation Algorithm and Search Method 126
Table 14-8: Indicator Domain Estimation Parameters 127
Table 14-9: Modeled Density Factors 128
Table 14-10: Model Comparison Descriptive Statistics 129
Table 14-11: Parameters used for Resource Pit Shell Generation 135
Table 14-12: Resource Cutoff Parameters 135
Table 14-13: Mineral Resource Statement for the Empire Mine, Custer County, Idaho, U.S.A., ... 136
Table 15-1: Recommended Inter-Ramp Slope Angles 137
Table 15-2: Reserve Pit Optimization Parameters 138
Table 15-3: Mineral Reserve NSR Cutoff 139
Table 15-4: Mineral Reserve Statement for the Empire Mine, Custer County, Idaho, U.S.A., ... 140
Table 16-1: Recommended Inter-Ramp Slope Angles 142
Table 16-2: Pit Design Criteria 144
Table 16-3: Yearly Mine Production Schedule 150
Table 16-4: Mine Schedule Parameters 152
Table 16-5: Equipment Availabilities 153
Table 16-6: Drill and Blast Parameters 154
Table 16-7: Load and Haul Parameters 155
Table 16-8: Mine Production Equipment 156
Table 16-9: Mine Support Equipment 156
Table 17-1: Project Design Criteria 157
Table 18-1: 4,000 stpd Waste Rock Storage Facility Parameters 169
Table 18-2: 4,000 stpd Tailings Storage Facility Parameters 171
Table 18-3: Plant Mobile Equipment 176
Table 20-1: List of Environmental Permits, Reviews and Approvals 181
Table 21-1: Operating Periods 191
Table 21-2: Estimated Capital Cost Summary (USD $000s) 192

Table 21-3: Mine Capital Cost Summary (USD $000s) 193
Table 21-4: Capital Cost Estimating Methodology – Process Plant and Ancillary Facilities 194
Table 21-5: Construction Cost Basis for the Processing Plant 195
Table 21-6: Construction Cost Basis Other Areas of the Capital Cost Estimate 195
Table 21-7: Cost Methodology for Other Areas of the Capital Cost Estimate 196
Table 21-8: Estimated Process Plant Capital Cost Summary (USD $000s) 200
Table 21-9: Estimated Project Services & Infrastructure Capital Cost Summary (USD $000s) 200
Table 21-10: Estimated Management, Engineering, and EPCM Services Capital Cost Summary (USD $000s) 200
Table 21-11: Pre-Production Costs Capital Cost Summary (USD $000s) 201
Table 21-12: Total Operating Cost Summary (USD) 201
Table 21-13: Mine Operating Costs (USD) 202
Table 21-14: Mine Operating Costs by Category (USD) 202
Table 21-15: Manpower Requirements for Mine Operations 203
Table 21-16: Estimated Processing Plant Operating Costs (USD) 204
Table 21-17: Estimated Process Plant Personnel and Labor Rates (USD) 207
Table 21-18: Process Reagents 208
Table 21-19: Process Consumables 208
Table 21-20: G&A Operating Costs (USD) 208
Table 21-21: G&A Operating Manpower 209
Table 22-1: Summary of Economic Results (USD) 210
Table 22-2: Metal Prices and Smelter Terms 211
Table 22-3: Metal Production and Net Revenue Estimate 211
Table 22-4: Estimated Capital Cost Summary (USD $000s) 212
Table 22-5: Total Operating Cost Summary (USD) 212
Table 22-6: Mine Operating Costs (USD) 213
Table 22-7: Process Operating Costs (USD) 214
Table 22-8: G&A Operating Costs (USD) 215
Table 22-9: Production Schedule and Cashflow Model 216
Table 22-10: Copper Price Sensitivity Economic Results 221
Table 23-1: Items for Procurement 231
Table 24-1: Risks Associated with Different Areas 234

DISCLAIMER

This report has been prepared using the NI 43-101 format but is not NI 43-101 compliant.

1. SUMMARY

1.1 Introduction

Konnex Resources, Inc. is a base metal exploration company engaged in the acquisition, exploration, and development of North American mineral properties. Konnex Resources, Inc. has retained Forte Dynamics (Forte) as primary author for this Pre-Feasibility Study with support from Hard Rock Consulting (HRC), and Valley Science and Engineering (VSE) for the company's principal property, the Empire Mine Project (the "Empire Project" or "Project"), a past-producing high-grade copper, gold, silver and tungsten property located in Custer County, Idaho, USA. Konnex Resources, Inc. is the Idaho registered operating company and is an 80% owned subsidiary of Phoenix Copper Limited (PXC) (AIM:PXC) (OTCQX:PXCLY), a private resource company incorporated in the British Virgin Islands. ExGen Resources, Inc (TSX V:EXG) (OTC:BXXRF), holds a 20% interest in Konnex. For the purposes of this report, PXC, and Konnex Resources, Inc. are referred to collectively as "Konnex".

1.2 Location

The Empire Mine Project is located within the Alder Creek Mining District of east central Idaho. This region lies to the east of the Idaho Batholith and north of the Snake River Basalt Plain, within the Cordilleran thrust belt at the northern edge of the Basin and Range structural province (Figure 1-1). The rock formations of the district comprise a thick series of Carboniferous limestones intruded by a batholithic mass of late Cretaceous granite to early Eocene porphyry, all traversed by narrow dikes of trachyte porphyry closely related to the granite porphyry in age and composition but differing from it markedly in general appearance. The Project area overlies a north-trending contact zone between an Eocene granitic complex, including the Mackay Granite and Mackay Porphyry, and the Upper Mississippian age White Knob Limestone. This contact zone includes a garnet-pyroxene-magnetite skarn developed in both the carbonate and intrusive rocks. The skarn hosts the polymetallic copper mineralization which characterizes the Empire Mine. The intrusive contact is sharp and dips steeply eastward.

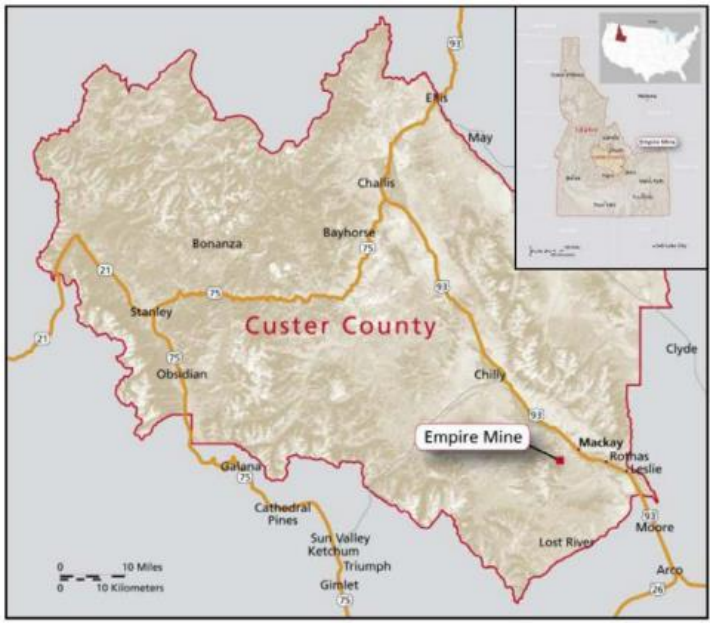


Figure 1-1: Empire Mine Project Location

1.3 Property Description

The Empire Project is in southeast-central Idaho, in the Alder Creek Mining District approximately 3.3 miles southwest of the town of Mackay, and 97 miles west of Idaho Falls. The Project area covers approximately 1,774 acres (718 hectares) of land surface within Section 12, T6NR23E, Sections 35 and 36, T7NR23E, and Sections 31 and 33, T7NR24E, Boise Prime Meridian. Approximately 95% of the Empire Mine deposit is located beneath patented (private) land surface. The remainder of the deposit is located within public lands administered by the U.S. Forest Service ("USFS").

The Project area consists of 109 contiguous mining claims, which together are comprised of the Honolulu Copper claim group, the Mackay LLC claim group, and the Horseshoe Claim Block. The Honolulu Copper group consists of 13 unpatented mining claims, 18 patented mining claims, and 5 unpatented mill site claims. The Mackay LLC group consists of 14 unpatented mining claims and 3 patented mining claims. Konnex holds 100% of the mineral rights to the claims via lease agreements with Honolulu Copper Corp. (Honolulu Copper group) and Mackay, LLC (Mackay group), with the exception of two Honolulu Copper group claims, for which Konnex controls a 50% share of the mineral rights. The Horseshoe Claim Block consists of 54 unpatented claims acquired in 2017 and 2018. The Horseshoe claims are 100% owned by Konnex.

The ownership of Konnex and the Empire Project is characterized by three agreements between ExGen and PXC: the Konnex Option dated July 15, 2015, the Supplemental Option Agreement dated November 9, 2016, and Supplemental Option Agreement No. 2 dated April 21, 2017. The terms of these agreements were fully satisfied as of June 29, 2017, establishing PXC's 80% ownership of Konnex Resources, Inc., and ExGen's 20% interest in Konnex.

1.4 Geology and Mineralization

The Empire Mine Project is located within the Alder Creek Mining District of east central Idaho, positioned east of the Idaho Batholith and north of the Snake River Basalt Plain. This district sits within the North American Cordilleran thrust belt at the northern edge of the Basin and Range structural province.

The Empire Project area overlies a north-trending contact zone between an Eocene granitic complex, including the Mackay Granite and Mackay Porphyry, and the Upper Mississippian age White Knob Limestone. This contact zone hosts a garnet-pyroxene-magnetite skarn developed in both the carbonate and intrusive rocks. The skarn is notably significant, serving as the host of the polymetallic copper mineralization characteristic of the Empire Mine. The intrusive contact is sharp and dips steeply eastward.

At the Empire Mine, both copper-oxide (carbonates, malachite and azurite) and sulphide (chalcopyrite/chalcocite) mineralization is developed to varying degrees within exoskarn in rafted limestone fragments and endoskarn in porphyry. The copper oxide mineralization occurs as veinlets, stockworks, and disseminated oxide/sulphides. The sulphides have similar characteristics, but also occur as massive lenses, both copper sulphides and magnetite, along skarn-hosted fault breccias. In both breccia types, the degree of mineralization appears to be a function of the amount of contained skarn fragments.

Drilling has identified a skarn-hosted body of disseminated and stockwork copper-oxide mineralization extending over a strike length of 4,000 ft, with a thickness of 6 to 240 ft from surface, and a width of up to 420 ft. The "width" figure is a function of topography; the skarn is exposed along a steeply inclined north-trending ridge-crest, with the northernmost outcrop being 840 ft lower in elevation than the southernmost exposure. All of the mineralized intercepts are in endoskarn, exoskarn and skarn-hosted breccias. The mineralization intersected is oxidized from surface to a vertical depth of approximately 400 ft, with sulfide

mineralization dominating below that depth. The transition zone between oxide and sulfide extends over tens of feet.

1.5 Mineral Resource Estimate

In the spring of 2020, the 2019 mineral resource estimate for the Empire Mine resource area with a new methodology that incorporates a geologic domain (versus grade domain) based estimate. This latest update incorporates of 356.4 ft of core drilling and 5,215 ft of RC drilling completed in 2020. The mineral resource estimate for the Red Star sulfide resource area was completed in 2019. No changes have been made to the estimate, so the associated effective date for Red Star mineral resource statement remains April 10, 2019 (HRC 2020).

The mineral resource estimate reported herein was prepared in a manner consistent with the Committee of Mineral Reserves International Reporting Standards (“CRIRSCO”), of which both the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) and Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the “JORC Code”) are members. The mineral resources are classified as Measured, Indicated and Inferred in accordance with “CIM Definition Standards for Mineral Resources and Mineral Reserves”, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014. Classification of the resources reflects the relative confidence of the grade estimates.

The geologic model for the Empire mineral resource estimate was created in two steps using Leapfrog V5.1.2. First, broad geologic domains were created from drillhole data for five (5) geologic units; The Mackay Granite, limestone, a general grouping of skarns (magnetite, iron oxide breccia, pyroxene and garnet), granite porphyry, and overburden. Following the construction of these domains, an indicator estimation methodology was used to estimate the individual lithologies (granites, limestone, iron oxide breccia, magnetite skarn, pyroxene skarn, garnet skarn, granite porphyry, and latite dikes) into the general skarn and granite porphyry domains using an inverse distance to the power of 2.5 with a maximum distance of 300 ft. Blocks were assigned a lithology based on the majority percentage from the indicator estimate. Blocks without an estimate were assigned a lithology from the broad geologic domains. Wireframes were created from the block model using Datamine Studio RM V1.1.20.0 and represent the final estimation domains.

The copper, gold, and silver grades were estimated from 20-foot down-hole composites using OK. Composites were coded according to the estimation domain. The search volumes were established based on practitioner’s experience with similar style deposits. The estimation was completed in a single pass with the maximum search volume set to 400 feet and using an approximate anisotropic ratio of 3:2:1. The same search volume was used to select samples for the mineral resource estimation for all metals in three domains, and an Inverse Distance (ID) to the power of 2.5 was used to estimate grade for all metals in a single domain. A true thickness composite length weighted ID to the power of 2.5 was used to estimate grade for the Red Star Sulfide Area.

The Mineral Resources for the deposit were evaluated using a Lerchs–Grossmann (L–G) pit optimizer to generate optimized pit shells.

Table 1-1 summarizes the cost and recovery parameters used in the analysis. Blocks classified as Measured, Indicated, and Inferred were used to define the resource pit shell (Table 1-2).

Table 1-1: Parameters Used for Resource Pit Shell Generation

Resource Parameter	Units	Cu	Au	Ag
Commodity Prices	**$/oz or $/lb**	**$4.25**	**$1,900**	**$25.50**
Flotation Process Recoveries				
Flotation _ Cu Concentrate	%	33.0%	50.0%	36.0%
Concentrate (Payables)				
Flotation_ Cu Concentrate (Au Payable based on grade)	%	95.0%	90-97%	95.0%
Cementation Process Recoveries				
Cementation (Total Copper Recovery after Flotation)	%	90.0%	0.0%	0.0%
Cementation Payable	%	100.0%	0.0%	0.0%
Treatment/Refining Charges				
Copper Con. Refining	Ag $/oz	0.40		
Copper Con. Refining	Au $/oz	4.00		
Copper Con. Trucking & Shipping $/t conc	wet	$80.00		
Copper Con. Treatment $/t conc	wet	$90.00		
Copper Cementation Shipping $/lb	Cu $/lb	$0.04		
Copper Cementation Selling $/lb	Cu $/lb	$0.02		
Operating Costs				
Mining Cost – Surface	$/t mined	$2.32		
Mining Cost - Incremental Increase for each 20ft depth	$/t mined	$0.016		
Processing Cost	$/t milled	$17.00		
G&A	$/t milled	$2.00		
Total Ore cost $/t milled	$/t milled	$19.00		
Pit Slope Assumptions	Five sectors were modelled based on core logging with inter-ramp angles ranging from 44º to 50º			

The mineral resource estimate for the Empire Mine resource area is summarized in Table 1-2. Mineral resources are not mineral reserves and do not have demonstrated economic viability such as diluting materials and allowances for losses that may occur when material is mined or extracted; or modifying factors including but not restricted to mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, and governmental factors. No existing environmental, permitting, legal, title, taxation, socio-economic, or other relevant factors that might materially affect the mineral resource estimate. Inferred mineral resources are that part of the mineral resource for which quantity and grade or quality are estimated on the basis of limited geologic evidence and sampling, which is sufficient to imply but not verify grade or quality continuity. Inferred mineral resources may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

Table 1-2: Mineral Resource Statement for the Empire Mine, Custer County, Idaho, U.S.A., Hard Rock Consulting, LLC, January 1, 2024

Classification	Tons	Copper		Gold		Silver		Copper Equiv.	
	(x1000)	%	lb (x1000)	opt	oz (x1000)	opt	oz (x1000)	%	lb (x1000)
Measured	8,467	0.48	80,643.61	0.012	98.9	0.41	3,497.4	0.67	113,590.89
Indicated	3,120	0.50	30,989.95	0.005	15.2	0.41	1,272.1	0.61	37,824.15
Measured + Indicated	**11,587**	**0.48**	**111,633.57**	**0.010**	**114.1**	**0.41**	**4,769.5**	**0.65**	**151,415.04**
Inferred	355	0.57	4,043.13	0.005	1.9	0.17	59.2	0.66	4,666.06

Notes:

(1) The effective date of the 2024 Mineral Resource estimate is January 1st, 2024. The QP for the estimate is Zachary J. Black, RM-SME, of Hard Rock Consulting, LLC.

(2) Mineral resources are not mineral reserves and do not have demonstrated economic viability.

(3) Mineral Resources are reported inclusive of Mineral Reserves at $19.00 NSR cut-off. The Mineral Resources are considered amenable to open pit mining and are reported within an optimized pit shell. The pit optimization is based on total ore costs of $19.00/t processed, mining costs of $2.32/t at surface and increasing $0.016/t for every 20ft of depth and pit slope angles ranging from 44 to 50 degrees.

(4) Cu-Eq (copper equivalent grade) is based on the mill recoveries and metal prices shown in Table 14-11.

(5) Copper Equivalent (CuEq) = ((Cu head grade x recovery x Cu Price) + (Au head grade x recovery x Au Price) + (Ag head grade x recovery x Ag Price)) / (Cu recovery x Cu Price)

(6) Tonnage and grade estimates are in Imperial units.

(7) Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

1.6 Mining Methods

Since the Empire Mine contains mineralization at or near the surface, open-pit mining method will be utilized to provide feed to the 4,000 tpd flotation mill followed by cementation plant to leach and recovery copper.

The life-of-mine strip ratio will be 1.69:1 and 1.1 million tons will be processed during the eight-year mine life.

The open-pit mining will employ two 7.5 yd^3 front end loaders and 51x45 ton articulated haul trucks to transport ore to the crushing plant and waste material to the designated rock waste dump.

The mine will operate seven days a week with two 12-hour shifts per day. Four crews are planned to cover the rotating schedule.

1.7 Metallurgy

The metallurgy test work has resulted in the development of a process flowsheet consisting of three-stage crushing circuit to prepare the ore for grinding in a closed-circuit ball mill with cyclones to produce flotation feed with a P_{80} of 150 microns. The flotation feed is sulfidized to recover oxide copper along with sulfide copper and gold and silver into a concentrate which is cleaned twice to produce marketable copper concentrate. The sulfide tailings are subjected to carbonate flotation to remove acid consumers, and the tailings are sent to acid leach circuit for recover of remaining copper oxide using the cementation process. Figure 1-2 provides the schematic diagram of the flowsheet.

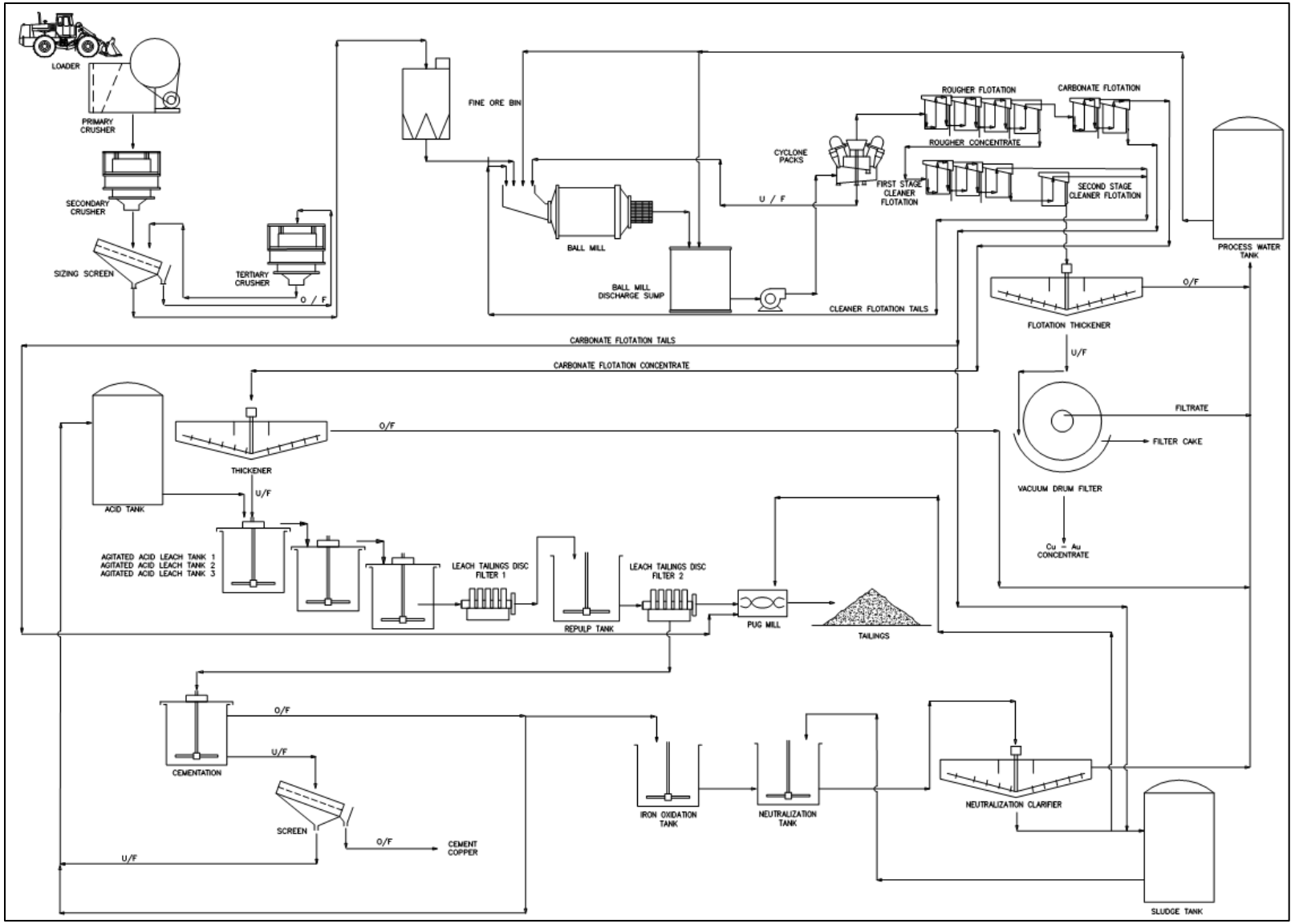


Figure 1-2: Overall Process Flowsheet

1.8 Mineral Processing

The detailed design criteria was developed for the process plant and is given in Section 17.

The plant was designed to process 4,000 stpd with 92% available and 350 dpy. The plant design feed grade will be variable, but the average feed grade will be 0.5% Cu and 0.5 g/t Au.

The products will be produced in the plant, namely copper concentrate containing gold and cemented copper.

1.9 Project Infrastructure

Access to local resources and infrastructure is excellent. The Project is located adjacent to the city of Mackay to allow for an easy commute for workers. The area has been historically mined and offers the necessary support functions that are found in a medium-sized city with regard to supplies, accommodations, communications, etc. The property has power and water utilities located adjacent to the property. Powerlines do traverse the property for legacy operations but will need to be upgraded for proposed operation.

Planned infrastructure for the site includes the following:

- Gate House, truck scales;
- Ammonium Nitrate and Fuel Oil (ANFO) Facility;
- Truck Maintenance Pad (Heavy Vehicle (HV) Workshop, Lube Farm, Washdown and Tire Change, Warehouse, Fuel Farm, Mining Offices);
- Crushing Process consisting of Primary, through tertiary;
- Concentrator consisting of (Ball Mill, Flotation and Leaching Process);
- Dry /Stack Tailings Storage Facility;
- Power Supply;
- Communications;

1.10 Market Studies and Contracts

The operation will produce marketable-grade copper concentrate containing precious metals (gold and silver) and cemented copper.

1.10.1 Markets

The traders will buy the copper concentrate due to it containing precious metals and no penalty elements with concentrates bought from other small operations.

1.10.2 Contracts

No contracts are in place as of the completion of the PFS. However, it is recommended that discussions with smelters and/or traders be initiated prior to the completion of the FS and/or start of construction.

1.11 Social and Environmental Aspects

Project permitting will commence upon submittal of a Project Plan, consisting of a Reclamation Plan and Operations Plan, to the Idaho Department of Lands (IDL) for review and approval under the Idaho Joint Review Process (IJRP). Approval of construction on federal public lands will be subject to environmental review under the National Environmental Policy Act (NEPA). The Empire Mine is a brownfield development of a historic mine, with no known liabilities or pending enforcement actions. Baseline environmental studies have been completed for submittal to permitting agencies with no environmental resource limitations identified that would limit project development in accordance with regulations and applicable environmental protection measures. A preliminary reclamation plan has been developed that meets regulatory requirements and industry standards for site stabilization, erosion control, and water resources protection.

The project will provide social benefits including employment and tax revenue to the community. The increase in population during project execution is projected to have only a minor impact on demand for community utilities, social services and public safety. Konnex has been actively engaged with the community to provide project information and gather input to improve the project.

1.12 Capital and Operation Cost Estimates

1.12.1 Capital Cost Estimates

LoM capital requirements are estimated at $83.9 million as summarized in Table 1-3. Initial capital of $62.5 million is required to commence operations. At the end of operations, the Project will not receive credit for asset sales and salvage.

Table 1-3: Estimated Capital Cost Summary (USD $000s)

Capital Costs ($000s)	Initial	Sustaining	Total LOM
2000 Mine Infrastructure and Equipment	$6,611	$16,409	$23,020
3100 Crushing and Conveying	$4,327	$0	$4,327
3200-3400, 3520 & 3530 Process Plant	$22,766	$0	$22,766
4000 Infrastructure and Utilities	$9,728	$0	$9,728
3540 Tailings and Ponds	$9,742	$6,474	$16,216
Total Direct Costs	$53,173	$22,883	$76,056
8000 Owner Costs, EPCM, Commissioning	$6,760	$0	$6,760
9100 - 9400 First Fills, Spares	$2,666	$0	$2,666
8500 Reclamation & Bonding	$0	$1,300	$1,300
9700 Contingency	$0	$0	$0
10000 Salvage Value	$0	($2,903)	($2,903)
Total Indirect Costs	$9,427	($1,603)	$7,824
TOTAL Capital	**$62,600**	**$21,280**	**$83,880**
9500 Working Capital	$891	($891)	$0
TOTAL CAPITAL/Working Capital	**$63,491**	**$20,389**	**$83,880**

1.12.2 Operating Cost Estimates

1.12.2.1 Mine

The mining operating cost area summarized for this project in Table 1-4.

Table 1-4: Mining Operating Costs (USD)

Department	Average Yearly Costs	$/ton Mined	$/ton Moved
Mine G&A	$639,916	$0.17	$0.15
Drilling	$1,265,510	$0.34	$0.30
Blasting	$1,410,661	$0.37	$0.34
Loading	$1,225,039	$0.33	$0.29
Haulage	$2,610,524	$0.69	$0.62
Roads & Dumps	$1,789,726	$0.48	$0.43
Dewatering	$24,944	$0.01	$0.01
Mine Mtce.	$1,181,102	$0.31	$0.28
Engineering	$469,883	$0.13	$0.12
Geology	$599,703	$0.16	$0.14
Grand Total	**$11,217,007**	**$2.99**	**$2.69**

1.12.2.2 Concentrator

The concentrator operating cost estimates for this project are summarized in Table 1-5.

Table 1-5: Estimated Processing Plant Operating Costs (USD)

Operating Cost	Total Cost ($)	$/Short Ton	% of Total Cost
Labor	2,965,040	2.12	15.4%
Power	2,592,863	1.85	13.4%
Fuel	516,012	0.37	2.7%
General Maintenance, @3%	825,265	0.59	4.3%
Reagents	8,407,350	6.01	43.6%
Consumables	3,983,000	2.84	20.6%
Total	**19,289,530**	**13.78**	**-**

1.12.2.3 Total Operating Cost

The total operating cost for mining, processing (concentrator), G&A, licensing and tax, and also royalties. is presented in Table 1-6.

Table 1-6: Total Cash Operating Costs (USD)

Operating Costs	$/CuEq lb	$/ton Ore	$/ton Mined	$/ton Moved
Total Mining	$0.76	$8.05	$2.99	$2.69
Total Processing	$1.30	$13.78		
Total Site General & Administration	$0.18	$1.87		
Mine License & Property Tax	$0.03	$0.30		
Cash Operating Costs	**$2.27**	**$24.00**		
Royalties	$0.17	$1.81		
Total Cash Costs	**$2.44**	**$25.81**		

1.12.3 Financial Analysis

The economic analysis of the base case scenario for the Project uses metal prices of $4.45/lb for copper, $2,325/oz for gold and $27.25/oz for silver, approximate spot price April 15, 2024. The economic model shows a Pre-Tax Net Present Value @ 7.5% ("NPV-7.5") of $87.86 million using a $20.50/ton NSR cut-off, as well as a Pre-Tax Internal Rate of Return ("IRR") of 46.4%. Table 1-7 summarizes the projected Cashflow; Net Present Value at varying rates; Internal Rate of Return; years of positive cash flows to repay the negative cash flow ("payback period"); multiple of positive cash flows compared to the maximum negative cash flow ("payback multiple") for the Project on both After-Tax and Before-Tax bases.

Table 1-7: Summary of Economic Results (USD)

Project Evaluation Overview	After Tax	Before Tax
Cumulative Net Cashflow (millions)	$132.44	$152.98
NPV @ 5.0%; (millions)	$89.55	$105.44
NPV @ 7.5%; (millions)	$73.75	$87.86
NPV @ 10.0%; (millions)	$60.71	$73.29
Internal Rate of Return	40.2%	46.4%
Payback Period	1.66	1.41
Payback Multiple	2.92	3.21
Benefit Cost Ratio	7.61	8.87
Initial Capital (millions)	$62.60	$62.60
Max Neg. Cashflow (millions)	-$69.09	-$69.09

1.13 Conclusions and Recommendations

1.13.1 Feasibility Study

A feasibility study (FS) should be completed to advance the Project and provide detailed information to support capital and operating cost-estimates for a potential project development decision.

The estimated budget for the FS is approximately $1.5 million.

1.13.2 Geology, Resources, and Mine Planning

The oxidation state has not been systematically collected in the database from operator to operator and will need to be addressed. Konnex geologists are delineating the oxidation state in an effort to refine the model for use in more advanced studies.

1.13.3 Mineral Processing

1.13.3.1 Conclusions

The amount of metallurgical test work data developed from the Empire mine drill core samples has fed to the development of a robust process flowsheet for recovering copper, gold, and silver. Key conclusions from the metallurgy study are:

- A conventional process flowsheet consisting of crushing, grinding, and rougher and cleaner flotation will recover sulfidized coper oxide, sulfide copper and gold and silver in a saleable copper concentrate.

- The flotation tailing will be conditioned with reagents and carbonates are floated in order to minimize acid consumption for leaching of non-floatable copper oxides.
- Additional copper can be recovered by acid leaching followed by cementation.

1.13.3.2 Recommendations

The ongoing test work is directed towards optimization of the process parameters. The study should lead to reduction of reagent usage thereby reducing operating costs as well as improving gold, silver, and copper recoveries thereby enhancing revenue.

1.13.4 Infrastructure

The level of design provided for this Pre-Feasibility Study indicates that the site can effectively support the mining operation that has been proposed. There is comparable evidence of this with the legacy operation. The below recommendations included continued study and coordination on the critical elements for mining infrastructure, which includes access, water, power and subsurface geotechnical for foundations.

The mine access has some flexibility and options. The option included in this study reduces the number of improvements required to access the pit and process facility. As continued input is received from the general public, other options could be further evaluated for best fit for the project.

Continued coordination with the City of Mackay for water usage and details on tie in point will be required as the project progresses. Similar coordination will be required with the Lost River Electric Company.

A field geotechnical program will be required in future stages of project study to obtain and or verify foundation characteristics for proposed infrastructure.

1.13.5 Environmental and Social Impacts

A Project Plan will be developed and submitted to the permitting agencies. Additional geochemical testing will be performed on tailings samples.

1.13.6 Results of the Site-Wide Water Balance Model

A hydrologic baseline study report will be completed that updates to the monitoring data reported in 2021 and incorporates groundwater characterization information (geological logging of boreholes, aquifer testing, and geophysics), as well as watershed-scale water balance models, into a conceptual site model for the project.

1.13.7 Groundwater Hydrology and Mine Dewatering

No further groundwater hydrologic studies are recommended at this time. Based on the relatively shallow depths of the proposed open pit mines and work to date, the mining activities will not intercept any significant groundwater requiring dewatering.

1.13.8 Process Plant Geotechnical Investigation

Geotechnical work is recommended during the FS study, particularly for foundation design of the processing facilities.

1.13.9 TSF Design

As part of advancing the TSF design in the FS, work should include optimization of the TSF construction schedule, geotechnical investigation, and assessment, TSF water balance update, and TSF consequence classification. The estimated budget for this work is included in the FS budget estimate.

2. INTRODUCTION

2.1 Background Information

The Empire Project is located in southeast-central Idaho, in the Alder Creek Mining District approximately 3.3 miles southwest of the town of Mackay, and 97 miles west of Idaho Falls. The Project area covers approximately 5,717 acres (2314 hectares) of land surface within Section 1, T6NR22E; Sections 1 and 2, T6NR23E; Section 6, T6NR24E, Sections 25, 26, 35, and 36, T7NR22E; Sections 20, 21, 25 through 32 and 34 through 36, T7NR23E; and Sections 30 through 32, T7NR24E, Boise Prime Meridian. Approximately 95% of the Empire Mine deposit is located beneath patented (private) land surface. The remainder of the deposit is located within public lands administered by the U.S. Forest Service (USFS), and to a lesser extent the Bureau of Land Management (BLM). The Project area is comprised of a combined 307 patented and unpatented mining and millsite claims. Prior to 2017, the Project area was limited to 55 contiguous mining claims, which together are comprised of the Honolulu Copper claim group and the Mackay claim group. The Honolulu Copper group consists of 13 unpatented mining claims, 18 patented mining claims, and 5 unpatented mill site claims. The Mackay group consists of 14 unpatented mining claims and 3 patented mining claims. Konnex holds 100% of the mineral rights to the claims via lease agreements with Honolulu Copper Corp. (Honolulu Copper group) and Mackay, LLC (Mackay group), with the exception of two Honolulu Copper group claims, for which Konnex controls a 50% share of the mineral rights. The claim area was expanded in 2017 to include 54 unpatented lode claims covering the northern extension of known mineralization through to the old Horseshoe lead/zinc/copper mine, and another 4 claims to the south of the existing pit. In early 2019, Konnex added 194 unpatented claims to the north (Windy Devil) and west (Navarre Creek) of the main claim block. Konnex holds 100% of the mineral rights for all claims added in 2017 and 2019.

The ownership of Konnex and the Empire Project is characterized by three agreements between ExGen and Phoenix: The Konnex Option dated July 15, 2015, the Supplemental Option Agreement dated November 9, 2016, and Supplemental Option Agreement No. 2 dated April 21, 2017. The terms of these agreements were fully satisfied as of June 29, 2017, establishing Phoenix's 80% ownership of Konnex Resources, Inc., and ExGen's 20% interest in the Project.

2.2 Terms of Reference and Purpose of the Report

Acronym / Abbreviation	Definition
AASHTO	American Association of State Highway and Transportation Officials
ASCE	American Society of Civil Engineers
ASTM	ASTM International
CN	Curve Number
CQA	Construction Quality Assurance
Forte	Forte Dynamics, Inc
HRC	Hard Rock Consulting, LLC.
HDPE	High Density Polyethylene
IBC	International Building Code
Konnex	Konnex Resources, Inc.
NEHRP	National Earthquake Hazard Reduction Program
NOAA	National Oceanic and Atmospheric Administration
P&ID	Piping and Instrumentation Diagram
PFDS	Precipitation Frequency Data Server
PFD	Process Flow Diagram
PSD	Particle Size Distribution
ROM	Run of Mine
SCS	Soil Conservation Service
USGS	United States Geological Survey
VSE	Valley Science and Engineering
WPCP	Water Pollution Control Permit
WRSA	Waste Rock Storage Area

2.3 Sources of Information

A portion of the background information and technical data presented in this report was obtained from the following documents:

Hard Rock Consulting, 2019. Empire Mine Project; NI-43-101 Technical Report.

Malhotra, 2024. Empire Met Study Phase I Report.

Malhotra, 2024. Empire Met Study Phase II Report.

2.4 Units of Measure

	Imperial Customary		Metric	
	Units	Description	Units	Description
Time	y	year	y	year
	d	day	d	day
	h	hour	h	hour
	min	minute	min	minute
	s	seconds	s	seconds
Length	ft	feet	m	meter
	in	inch	cm	centimeter
	mil	one thousandth of an inch	mm	millimeter
	mi	miles	µm	microns
Area	ft^2	square feet	m^2	square meters
Mass	st	short ton	mt	metric tonne
	dst	dry short tons	dmt	dry metric tonnes
	kst	thousand dry short tons	kmt	thousand dry metric tonnes
	lb	pound	kg	kilogram
	oz	Troy ounce	g	gram
Grade	opt	Troy ounces per ton	gpt	grams per tonne
Volume	ft^3	cubic feet	m^3	cubic meter
	gal	gallons	L	liter
Density	lb/ft^3	pounds per cubic foot	t/m^3	tonnes per cubic meter
	sg	specific gravity	sg	specific gravity
Percent Solids	wt%	percent solids by weight		
Hardness	kWh/st	kilowatt-hours per short ton	kWh/t	kilowatt-hours per tonne
Elevation – Above Mean Sea Level (AMSL)	FASL	feet above sea level	MASL	meters above sea level
Volumetric Flow rate	gpm	gallons per minute	Lpm	liters per minute
	scfm	standard cubic feet per minute	m^3/hr	cubic meters per hour
Throughput	stph	short tons per hour	mtph	metric tonnes per hour
	stpy	short tons per year	mtpy	metric tonnes per year
Temperature	˚F	degrees Fahrenheit	˚C	degrees Celsius
Concentration	ppm	parts per million	mg/L	milligrams per liter
			g/L	grams per liter
Work Index	kWh/st	kilowatt hour per short ton	kWh/mt	kilowatt hour per metric tonne
Power	hp	horsepower	kW	kilowatt
Mill Speed	rpm	revolutions per minute	rpm	revolutions per minute
Pressure	psi	pounds per square inch	kPa	kilopascal

2.5 Detailed Personal Inspections

2.5.1 Open-Pit Surface Mine

The current mine workings include historic underground and open-pit mining, which have been in place for more than one hundred years and fifty years, respectively. The open-pit is located within the private land and can only be accessed through a security gate. Signs are posted declaring the property off-limits to the general public, and fencing is used for the same purpose. Safety considerations require the open pit excavations to be stabilized by best management practices to minimize rock fall. The following measures will be conducted as necessary to ensure stable and safe working:

- Regular inspection of the main haulage ways to identify any potential rockfall issues or concerns.
- Documentation of all inspections conducted, including details on issues addressed; and stabilization measures employed to ensure rock caving or movement will not affect future mining or create safety issues.
- Appropriate fencing and signage limiting entrance to the open pit to authorized personnel only.

2.5.2 Dry Stacked Tailings

The Dry Stacked Tailings Facility is located within private land. This proposed facility will have limited stabilization issues due to engineering design and the best management practices to control material sloughing, such as sloped surfaces to encourage runoff. Regular inspections of the Dry Stacked Tailings facility will be undertaken to identify any potential stability issues per IDEQ's regulations. Stability inspections will continue, as deemed necessary to ensure stability for the life of the project. If areas are identified, they will be stabilized by the most appropriate means.

2.5.3 Waste Rock Storage Facility

Two primary considerations associated with the WRSF include stormwater diversion and dust control. Stormwater diversions will be installed during the development of the WRSF. Dust control measures will be implemented on travel routes to and from the WRSF. The following actions will be undertaken, as necessary, in compliance with state and federal regulations:

- Monitoring for visible particulate matter (dust and emissions) as required by the mine's air quality permit.
- Daily monitoring for structural issues and pond level as following management practices based on federal, state, and local regulations.
- Annual inspection by a certified engineer for structural issues and stormwater control effectiveness following management practices based on federal, state, and local regulations.
- Visual assessments after rain events of stormwater discharged from outfalls below the barren rock stockpiles following management practices based on federal, state, and local regulations.

2.5.4 Measures to Isolate or Control Toxic or Deleterious Materials

Any hazardous substances utilized on the mine property, including sulfuric acid, antifreeze, and spent petroleum products, will be managed per federal and state regulatory requirements through proper removal and disposal, as necessary. Monthly inspections of areas where petroleum products are stored will be completed following best management practices. In the event of a release, measures defined by federal, state, and local regulations will be followed as described. The following actions will be implemented to manage toxic materials:

- All unnecessary maintenance items, lubricants, and mining materials will be removed from the pit, and all items returned to the appropriate vendor or placed in proper on-site storage.
- All explosives and ANFO will be stored in appropriate on-site facilities located on private land.
- Upon termination of operations, the process solutions will be detoxified by rinsing to reduce harmful materials in the process water to a level meeting water standards.

2.5.5 Plans for Monitoring Site Conditions During Periods of Non-Operation or Curtailment

In compliance with applicable local, state, and federal regulations, Konnex will monitor site conditions during non-operation or curtailment as follows:

- Security will be monitored with cameras from the three guardhouses and any other cameras associated with buildings. Anomalies picked up by the cameras will be investigated by a site security guard who will also be responsible for the safety of the mine.
- All required environmental and inspection activities will continue to be performed, records maintained, and compliance reports filled to maintain compliance and avoid unnecessary and undue environmental degradation.
- A health and safety representative will visit the mine regularly to maintain workplace safety systems, provide safety training, conduct MSHA inspections, and complete compliance reporting.
- The Mine Operation’s Manager will provide overall site management with assistance from the operations and maintenance staff.

2.5.6 Procedure for Anticipated Temporary Closures

In the event of a temporary or planned closure, a temporary closure notice will be provided to the following regulatory agencies.

- Field Manager, BLM Challis District Office, 721 East Main Ave., Challis, Idaho 83226
- Field Manager, USFS Salmon-Challis National Forest Lost River Range District 716 W Custer St. Mackay, Idaho 83251
- Idaho Department of Environmental Quality, Water Quality Division and Air Quality Division, 1410 N Hilton St, Boise, ID 83706
- MSHA Mine Inspector, 300 E Mallard Dr, Boise, ID 83706

Updates of the temporary closure status will be provided to the BLM's and USFS's Field Office Managers on a semi-annual basis after that and 30 days before resuming mining operations.

2.5.7 Procedures to Resume Mining Operations

Konnex will maintain all aspects of the operation in good standing order, including equipment, the processing areas, and local, state, and federal permits in the case of temporary closure. Within 30 days of a decision to resume operations, Konnex will notify BLM, USFS, the Mine Inspector, and IDEQ of its intention to resume operations. A schedule facility for recommissioning will be provided. Existing permits will be complied with during the re-start process.

2.5.8 Past Inspections

In 2005, DE conducted a preliminary assessment and site inspection (PA/SI) of historical features at the Empire Mine (Idaho Department of Environmental Quality, 2005). The features included the Cossack Tunnel, Empire Mill, Alberta Tunnel, tramway "headhouse", Bullion Tunnel. And Blue Bird workings. The inspection identified six potential hazardous waste violations at the mill, including unlabeled containers, soil staining with oil and diesel fuel, and several lead-acid batteries that leaked. The EEQ also identified high arsenic concentrations in samples from an ore concentrate stockpile and tailings impoundment at the mill. The report indicated arsenic concentrations could be a potential risk to human health through intermittent exposure of recreational visitors by incidental ingestion, inhalation of particulates, and dermal contact. The report indicated that aquatic life is not supported by the ephemeral drainages within or near the site. Groundwater exposure pathways were identified as unlikely.

3. RELIANCE ON OTHER EXPERTS

HRC has fully relied upon and disclaims responsibility for information provided by Konnex regarding property ownership, mineral tenure, and permitting and environmental aspects of the Empire Project. Such information is presented in Section 4 of this report. Property title and mineral tenure was provided by personal communication with Ryan McDermott, CEO of Konnex, on May 29, 2019, and in written format via the following documents:

- Konnex Option Agreement, July 15, 2015
- Supplemental Option Agreement, November 9, 2016
- Supplemental Option Agreement No. 2, April 21, 2017
- Opinion as to Condition of Title – Mining Claims, May 4, 2019 (David P. Claiborne, Sawtooth Law Offices LLC)

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Empire Project is located in southeast-central Idaho, in the Alder Creek Mining District approximately 3.3 miles southwest of the town of Mackay, and 97 miles west of Idaho Falls (Figure 4-1). The project area covers approximately 5,717 acres (2314 hectares) of land surface within Section 1, T6NR22E; Sections 1 and 2, T6NR23E; Section 6, T6NR24E; Sections 25, 26, 35, 36, T7NR22E; Sections 21, 21, 25 through 32 and 34 through 36, T7NR23E; and Sections 30 through 32, T7NR24E, Boise Prime Meridian. The approximate geographic center of the Project is 43°53'N latitude and 113°40'W longitude. Topographic map coverage of the Project area is provided by the 1:24,000-scale, Mackay Reservoir 7.5-minute U.S.G.S. Topographic Quadrangle.

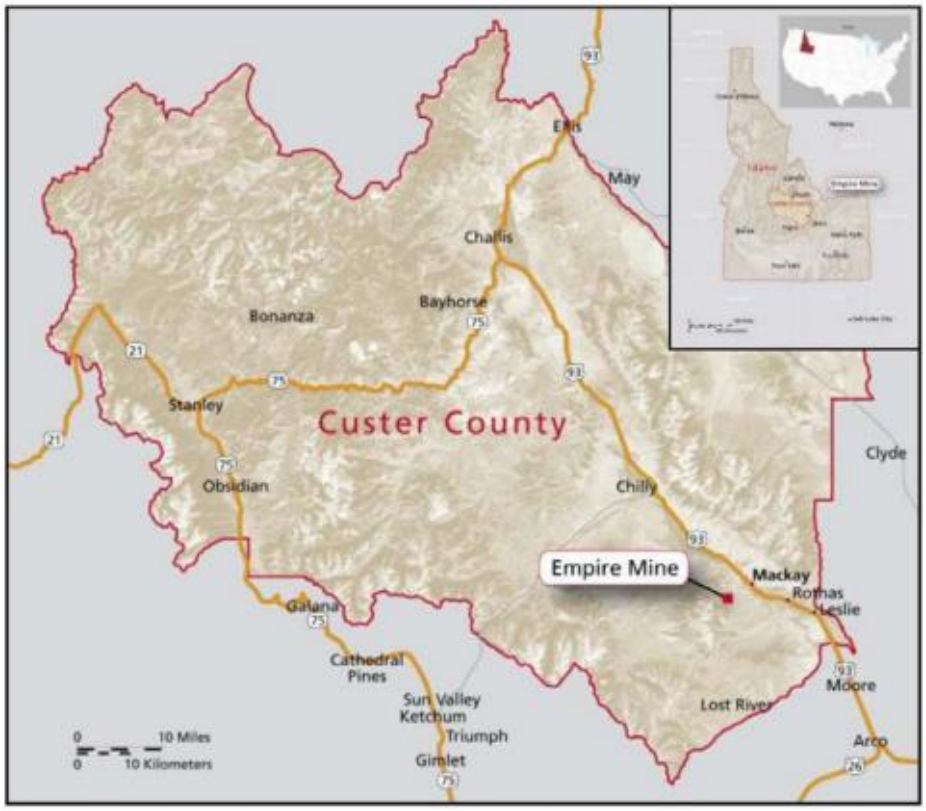


Figure 4-1: Empire Mine Project Location

The ownership of Konnex and the Empire project is characterized by the three agreements between ExGen and PXC: the Konnex Option dated July 15, 2015, the Supplemental Option Agreement dated November 9, 2016, and Supplemental Option Agreement No. 2 dated April 21, 2017 (collectively the "Option Agreements").

The Option Agreements allowed Phoenix to acquire 80% of the common shares of Konnex, ExGen's wholly owned subsidiary which holds the leases to the Empire Mine Project, on the following terms and conditions, including a term requiring the return of the Konnex common shares to ExGen in certain circumstances:

Project Participation

- Upon the deposit of Phoenix of US $1,000,000 into the Konnex bank account (discussed below under Project Expenditures by Phoenix), 80% of Konnex's common shares were transferred to Phoenix.
- ExGen to retain a 20% carried interest until commencement of mine construction.
- ExGen to be granted 2.5% net smelter returns royalty for all metals on the Empire Mine Project (the "**2.5% NSR**").
- 30-mile area of interest, which applies to both ExGen's 20% carried interest and the 2.5% NSR.
- If any of the cash or share payments, or project expenditures requirements, both as described below, are not completed as required pursuant to the Options Agreements, or if the Option Agreements are terminated, then the 80% of the Konnex common shares will be returned to ExGen without ExGen paying any consideration.

Cash and Shares

- ExGen was paid a cash payment of US $50,000 on signing the Original Option (PAID).
- ExGen was issued 5,000,000 common shares of Phoenix on signing the Original Option (ISSUED).
- ExGen was paid a cash payment of US $50,000 within 60 days of signing the Original Option (PAID).
- ExGen was paid a cash payment of US $50,000 on signing the Amendment (PAID).
- ExGen was issued an additional 5,000,000 common shares of Phoenix (substantially pursuant to the Original Option Terms) and an additional 1,300,000 common shares of Phoenix on signing the Amendment (ISSUED).
- ExGen to be paid cash payment of US $100,000 on the earlier of the Phoenix IPO date or by March 31, 2017 (PAID).
- ExGen to be paid US $100,000 on each anniversary date of the earlier of the Phoenix IPO or March 31, 2017, until the completion of a bankable feasibility study on the Empire Mine Project.
- The IPO Anniversary Payment increases 100% to US $200,000 for any payment where during the prior 12 months period the minimum expenditures on the Empire Mine Project has not been met (please see below for minimum expenditure requirements).

Project Expenditures by Phoenix

- Phoenix to have deposited a minimum of US $1,000,000 into the Konnex bank account by the earlier of the Phoenix IPO date or by June 30, 2017.
- Phoenix to spend the US $1,000,000 on the Empire Mine Project within 12 months of deposit into the Konnex bank account.
- Phoenix to fund all Empire Mine Project property maintenance and sustaining costs of Konnex.

- Phoenix to spend a minimum of US $500,000 on the Empire Mine Project every 12 months until the completion of a bankable feasibility study.

Deal Protection and Corporate Structure

- Should Phoenix sell its 80% interest in Konnex prior to the commencement of commercial production, ExGen shall have the right but not the obligation to either sell its 20% interest in Konnex on the same terms as Phoenix. Alternatively, ExGen may elect to have any acquiring party fund all ExGen's *pro rata* share of project capital costs by way of loan from Konnex, with interest payable by Konnex, without dilution to ExGen's 20% joint venture interest.

4.2 Mineral Tenure, Agreements, and Encumbrances

The Project area is comprised of a combined 307 patented and unpatented mining and millsite claims. Prior to 2017, the Project area was limited to 55 contiguous mining claims, which together are comprised of the Honolulu Copper claim group and the Mackay claim group. The Honolulu Copper group consists of 13 unpatented mining claims, 18 patented mining claims, and 5 unpatented mill site claims. The Mackay group consists of 14 unpatented mining claims and 3 patented mining claims. Konnex holds 100% of the mineral Konnex Resources Inc. NI 43-101 Technical Report Empire Mine Project Executive Summary November 25, 2020, 2 rights to the claims via lease agreements with Honolulu Copper Corp. (Honolulu Copper group) and Mackay, LLC (Mackay group), with the exception of two Honolulu Copper group claims, for which Konnex controls a 50% share of the mineral rights. The claim area was expanded in 2017 to include 54 unpatented lode claims covering the northern extension of known mineralization through to the old Horseshoe lead/zinc/copper mine, and another 4 claims to the south of the existing pit. In early 2019, Konnex added 194 unpatented claims to the north (Windy Devil) and west (Navarre Creek) of the main claim block. Konnex holds 100% of the mineral rights for all claims added in 2017 and 2019. Pertinent information for all claims, including name and serial/patent number, is tabulated in Table 4-1.

Table 4-1: Empire Mine Project Mining Claims

Claim Name	Serial/Patent No.	Type	Claimant/Owner	Survey No.	Tenure
Antelope	IMC196738	Lode	Honolulu Copper Corp.		Lease
Artic	IMC174901	Lode	Mackay LLC		Lease
Atlantic	30112	Patented	Mackay LLC	1272A	Lease
Atlantic Millsite	-	-	Honolulu Copper Corp.		Excluded
Blue Jay	729417	Patented	Honolulu Copper Corp.	2855	Lease
Blue Jay No.1	80603	Patented	HCC and JC Patterson, 50/50	2842	Lack of Control
Blue Jay No. 2	809603	Patented	HCC and JC Patterson, 50/50	2842	Lack of Control
Buena Vista	443615	Patented	Honolulu Copper Corp.	1887	Lease
Catherine	IMC174819	Lode	Mackay LLC		Lease
Continental	443615	Patented	Honolulu Copper Corp.	1887	Lease
Copper Bullion	30112	Patented	Honolulu Copper Corp.	1272A	Lease
Cushing	IMC196739	Lode	Honolulu Copper Corp.		Lease
Decatur	IMC196748	Lode	Honolulu Copper Corp.		Lease
Eastern	30112	Patented	Honolulu Copper Corp.	1272A	Lease
Eastern MS	IMC198241	Millsite	Honolulu Copper Corp.		Lease
Empire	IMC196743	Lode	Honolulu Copper Corp.		Lease
Evans	IMC196742	Lode	Honolulu Copper Corp.		Lease
Farragut	IMC196741	Lode	Honolulu Copper Corp.		Lease
Gem	IMC196737	Lode	Honolulu Copper Corp.		Lease
General Lawton	IMC198237	Lode	Honolulu Copper Corp.		Lease
Hamilcar	IMC196740	Lode	Honolulu Copper Corp.		Lease

Claim Name	Serial/Patent No.	Type	Claimant/Owner	Survey No.	Tenure
Hannibal	443615	Patented	Honolulu Copper Corp.	1887	Lease
Khedive	443615	Patented	Honolulu Copper Corp.	1887	Lease
Leslie W	443615	Patented	Honolulu Copper Corp.	1887	Lease
Mars	IMC196747	Lode	Honolulu Copper Corp.		Lease
Midwinter	30112	Patented	Honolulu Copper Corp.	1272A	Lease
ML1	IMC200218	Lode	Mackay LLC		Lease
ML11	IMC200223	Lode	Mackay LLC		Lease
ML3	IMC200219	Lode	Mackay LLC		Lease
ML5	IMC200220	Lode	Mackay LLC		Lease
ML7	IMC200221	Lode	Mackay LLC		Lease
ML9	IMC200222	Lode	Mackay LLC		Lease
Occidental	443615	Patented	Honolulu Copper Corp.	1887	Lease
Oriental	IMC174818	Lode	Mackay LLC		Lease
Oriental MS	IMC198242	Millsite	Honolulu Copper Corp		Lease
Pacific	30113	Patented	Mackay LLC	1272A	Lease
Pacific Millsite	-	-	Honolulu Copper Corp		Excluded
Pasha	443615	Patented	Honolulu Copper Corp	1887	Lease
Pasha MS	IMC198240	Millsite	Honolulu Copper Corp		Lease
Porter	IMC196746	Lode	Honolulu Copper Corp		Lease
Remonization	443615	Patented	Honolulu Copper Corp	1887	Lease
Remus	IMC174820	Lode	Mackay LLC		Lease
Rio Vista	443615	Patented	Honolulu Copper Corp.	1887	Lease
Romulus	IMC174821	Lode	Mackay LLC		Lease
Stephen	IMC174822	Lode	Mackay LLC		Lease
Sultan	443615	Patented	Honolulu Copper Corp.	1887	Lease
Sultan MS	IMC198238	Millsite	Honolulu Copper Corp.		Lease
Sultana	443615	Patented	Honolulu Copper Corp.	1887	Lease
Sultana MS	IMC198239	Millsite	Honolulu Copper Corp.		Lease
Telephone	IMC174817	Lode	Mackay LLC		Lease
Wainwright	IMC196744	Lode	Honolulu Copper Corp.		Lease
Washington	IMC196745	Lode	Honolulu Copper Corp.		Lease
Wellington	443615	Patented	Honolulu Copper Corp.	1887	Lease
Western	30114	Patented	Mackay LLC	1272A	Lease
White Knob	IMC174816	Lode	Mackay LLC		Lease
Granite #1	IMC19899	Lode	Ausich Mines LLC		Lease
Granite #2	IMC19900	Lode	Ausich Mines LLC		Lease
Granite #3	IMC19901	Lode	Ausich Mines LLC		Lease
Grande Lode		Patented	Ausich Mines LLC	1052	Lease
HS-1	IMC219135	Lode	Konnex Resources Inc.		Owned
HS-2	IMC219136	Lode	Konnex Resources Inc.		Owned
HS-3	IMC219137	Lode	Konnex Resources Inc.		Owned
HS-4	IMC219138	Lode	Konnex Resources Inc.		Owned
HS-5	IMC219139	Lode	Konnex Resources Inc.		Owned
HS-6	IMC219140	Lode	Konnex Resources Inc.		Owned
HS-7	IMC219141	Lode	Konnex Resources Inc.		Owned
HS-8	IMC219142	Lode	Konnex Resources Inc.		Owned
HS-9	IMC219143	Lode	Konnex Resources Inc.		Owned
HS-10	IMC219144	Lode	Konnex Resources Inc.		Owned
HS-11	IMC219145	Lode	Konnex Resources Inc.		Owned
HS-12	IMC219146	Lode	Konnex Resources Inc.		Owned
HS-13	IMC219147	Lode	Konnex Resources Inc.		Owned
HS-14	IMC219148	Lode	Konnex Resources Inc.		Owned
HS-15	IMC219149	Lode	Konnex Resources Inc.		Owned
HS-16	IMC219150	Lode	Konnex Resources Inc.		Owned

Claim Name	Serial/Patent No.	Type	Claimant/Owner	Survey No.	Tenure
HS-17	IMC219151	Lode	Konnex Resources Inc.		Owned
HS-18	IMC219152	Lode	Konnex Resources Inc.		Owned
HS-19	IMC219153	Lode	Konnex Resources Inc.		Owned
HS-20	IMC219154	Lode	Konnex Resources Inc.		Owned
HS-21	IMC219155	Lode	Konnex Resources Inc.		Owned
HS-22	IMC219156	Lode	Konnex Resources Inc.		Owned
HS-23	IMC219157	Lode	Konnex Resources Inc.		Owned
HS-24	IMC219158	Lode	Konnex Resources Inc.		Owned
HS-25	IMC219159	Lode	Konnex Resources Inc.		Owned
HS-26	IMC219160	Lode	Konnex Resources Inc.		Owned
HS-27	IMC219161	Lode	Konnex Resources Inc.		Owned
HS-28	IMC219162	Lode	Konnex Resources Inc.		Owned
HS-29	IMC219163	Lode	Konnex Resources Inc.		Owned
HS-30	IMC219164	Lode	Konnex Resources Inc.		Owned
HS-31	IMC219165	Lode	Konnex Resources Inc.		Owned
HS-32	IMC219166	Lode	Konnex Resources Inc.		Owned
HS-33	IMC219167	Lode	Konnex Resources Inc.		Owned
HS-34	IMC223006	Lode	Konnex Resources Inc.		Owned
HS-35	IMC223007	Lode	Konnex Resources Inc.		Owned
HS-36	IMC223008	Lode	Konnex Resources Inc.		Owned
HS-37	IMC223009	Lode	Konnex Resources Inc.		Owned
HS-38	IMC223010	Lode	Konnex Resources Inc.		Owned
HS-39	IMC223011	Lode	Konnex Resources Inc.		Owned
HS-40	IMC223012	Lode	Konnex Resources Inc.		Owned
HS-41	IMC223013	Lode	Konnex Resources Inc.		Owned
HS-42	IMC223014	Lode	Konnex Resources Inc.		Owned
HS-43	IMC223015	Lode	Konnex Resources Inc.		Owned
HS-44	IMC223016	Lode	Konnex Resources Inc.		Owned
HS-45	IMC223017	Lode	Konnex Resources Inc.		Owned
HS-46	IMC223018	Lode	Konnex Resources Inc.		Owned
HS-47	IMC223019	Lode	Konnex Resources Inc.		Owned
HS-48	IMC223020	Lode	Konnex Resources Inc.		Owned
HS-49	IMC223021	Lode	Konnex Resources Inc.		Owned
HS-50	IMC223022	Lode	Konnex Resources Inc.		Owned
HS-51	IMC223023	Lode	Konnex Resources Inc.		Owned
HS-52	IMC223024	Lode	Konnex Resources Inc.		Owned
HS-53	IMC223025	Lode	Konnex Resources Inc.		Owned
HS-54	IMC223026	Lode	Konnex Resources Inc.		Owned
NCC-1 to NCC-121	IMC228326 - IMC228446	Lode	Konnex Resources Inc.		Owned
WDC-1 to WDC-73	IMC228447 - IMC228519	Lode	Konnex Resources Inc.		Owned

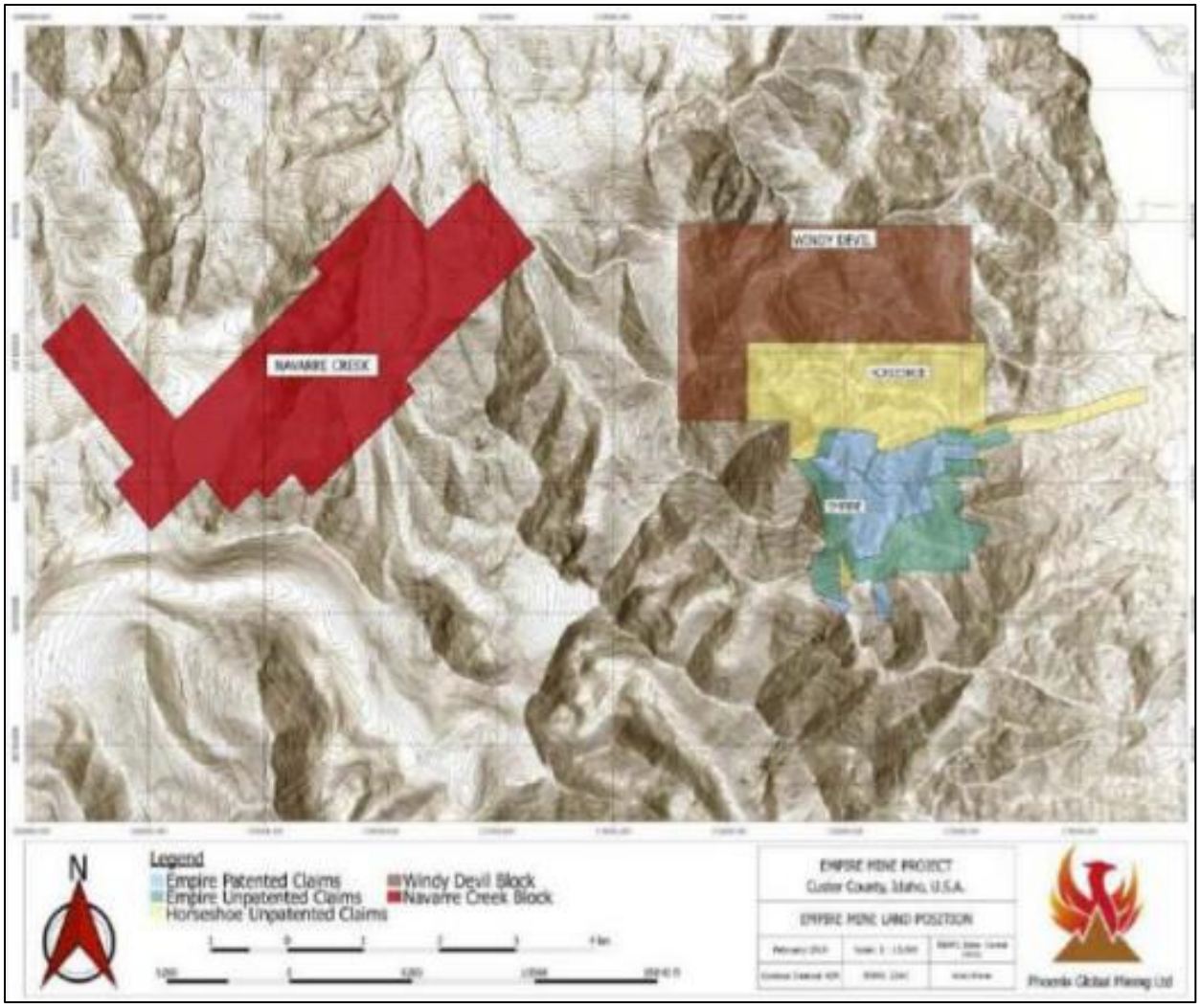


Figure 4-2: Empire Mine Project Claim Areas

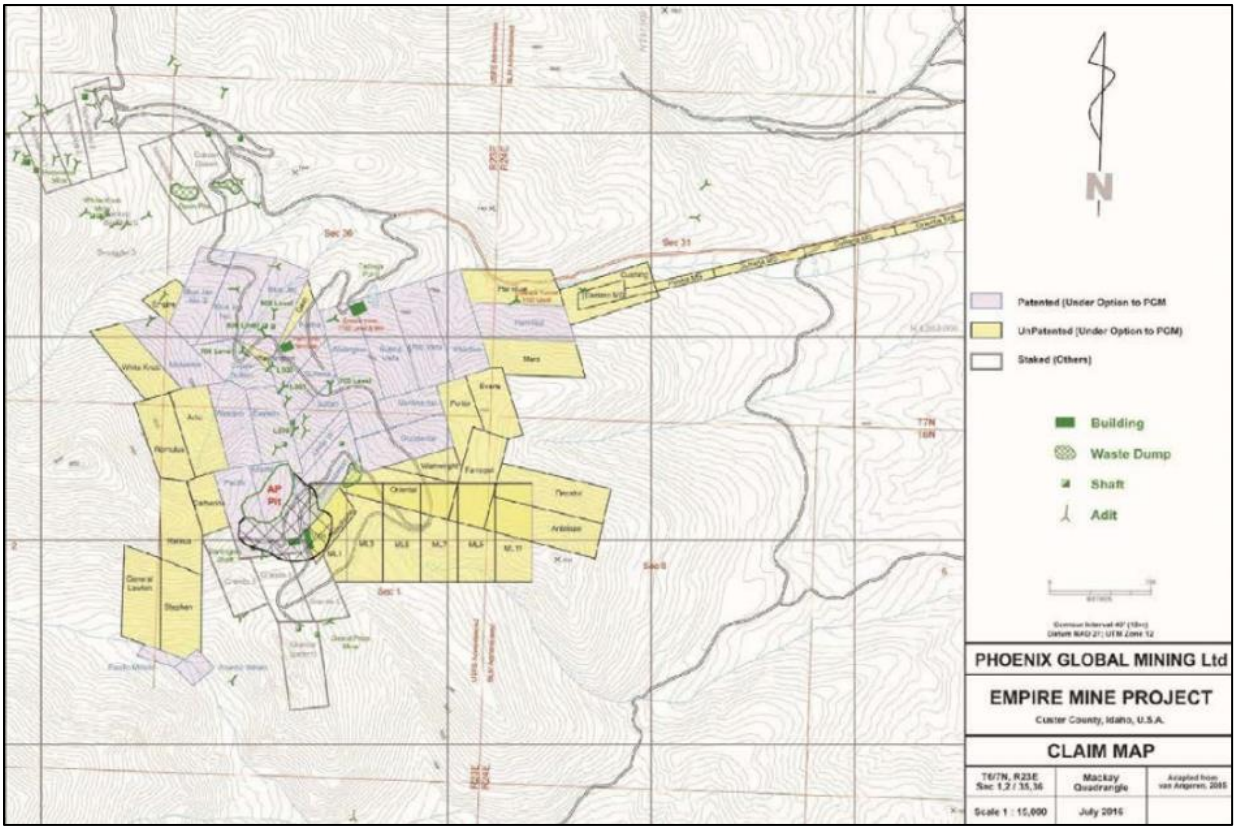


Figure 4-3: Empire Project Honolulu and Mackay Group Claim Areas

4.3 Risks

The oxidation state has not been systematically collected in the database from operator to operator and will need to be addressed. Konnex geologists are delineating the oxidation state in an effort to refine the model for use in more advanced studies.

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

5.1 Access and Climate

Primary access to the Empire Mine Project is provided by State Highway 93 East out of Idaho Falls, Idaho, for roughly 94 miles to the city of Mackay, and then 3.3 miles of well-maintained, all-weather gravel road from Mackay to the Project area. Access throughout the claim block, including to old workings and drill pads, is provided by an assortment of secondary gravel roads and jeep trails requiring four-wheel-drive or all-terrain vehicles.

The climate in the vicinity of the Project area is semi-arid, with long snowy winters and short, cool, dry summers. Maximum annual temperatures range from a high of 27.9°C in July to a low of -13.7°C in January. Precipitation occurs largely as spring rainstorms and winter snowfall. Total annual precipitation includes rainfall of about 9.7 inches and approximately 27.6 inches of snowfall.

5.2 Local Resources and Infrastructure

The community nearest to the Project area is the city of Mackay, which hosts a population of about 550. Mackay offers standard municipal amenities including lodging and services, and a limited supply of foodstuffs and hardware. The nearest major supply center is Idaho Falls, roughly 100 miles east of the Project area. Commercial air and rail service are both available in Idaho Falls, which is served by the Idaho Falls Regional Airport and the Eastern Idaho Railroad. Rail access is also available in the communities of Blackfoot and Pocatello, roughly 30 and 50 miles south of Idaho Falls, respectively. Ample skilled and unskilled labor can be found in Mackay, Idaho Falls, and a variety of other communities throughout the regional area.

A water right granted to Honolulu Copper Corporation with a priority date of June 1, 1884, was decreed for diverting up to 0.75 cubic feet per second (approximately 336 gallons per minute) from Cliff Creek to the Project area for mining consumptive use throughout the year and is currently considered for contingency only. The primary source for water will be from the City Spring located adjacent to the property where all water demand will be supplied. A distribution line owned by the Lost River Electric Cooperative will extend from a substation in Mackay to the Golf Course near the river, where Konnex will tie into the distribution line and serve the Mill and project. The route from the Golf Course to the private property will be aligned with a legacy overhead power alignment. Existing surface rights are sufficient to support all proposed exploration and mining activities, including tailings and waste storage areas and processing facilities.

5.3 Physiography

The Empire Mine Project area is situated along the north-eastern edge of the White Knob Mountains, on the eastern flank of Mackay Peak, at elevations ranging from 6200 ft to 9100 ft above mean sea level. Local terrain is generally steep and rugged, with numerous ridges and gullies. Northern slopes are sparsely to densely forested with a mixture of Doug fir, Poderosa and Lodgepole pine, and Engelmann spruce, while the southern slopes are generally open with a low cover of scrubby sagebrush and grasses.

6. HISTORY

6.1 History of Previous Exploration

A variety of publicly available documents exist which describe the early history of the Empire Mine in greater detail than is presented here. The reader is directed to “History of Selected Mines of the Alder Creek Mining District, Custer County, Idaho” (Mitchell, 1997) and “Geology and Ore Deposits of the Mackay Region, Idaho” (Umpleby, 1917) for a more thorough discussion of the early history of the Project.

The first significant advancement of the Empire Mine was accomplished by the Empire Copper Company between 1907 and 1921. The following paragraphs describe the mine and associated development work as of 1917 (Umpleby, 1917):

> “The Empire group of claims lies on the steep mountain side 3.5 miles southwest of Mackay, where the company's smelter is situated. The mine may be reached from Mackay either by wagon road or by a railroad owned by the Empire Copper Co. The railroad accomplishes the rise of 2,000 feet to the mine by a circuitous route 7.75 miles in length. It is equipped with two 23-ton Shay mountain-climbing locomotives and 38 cars.
>
> Development at the mine comprises between 20,000 and 25,000 feet of underground work. There are four principal groups of workings- the Darlington shaft, the Alberta tunnel, the Copper Bullion tunnel, and the Cossack tunnel. Of these, the Darlington shaft, 700 feet deep, is no longer accessible, and the Cossack tunnel, now 1,900 feet long, is still 2,000 feet from a point beneath the north Alberta shoot. This tunnel enters the hill from the northeast at an elevation of about 6,760 feet.
>
> The Copper Bullion tunnel, situated at an elevation of 7,610 feet, is about 1,600 feet long, and its laterals, raises, and winzes total perhaps 800 feet more. The Alberta tunnel, at an elevation of 7,700 feet, which comprises a main adit 2,800 feet long connecting with the Darlington shaft, and laterals totaling about 3,000 feet, is the most important single piece of development. At 400 feet above it is tunnel No. 300, which is approximately 1,000 feet long. Directly above this tunnel, at an elevation 125 feet higher, is the North tunnel. Southward around the hill from the North tunnel at elevations between 8,200 and 8,400 feet are several tunnels, chief among which are the Davis, Hunter, South, Starlight, Sunlight, Iron, and Quarry tunnels, each representing from 100 to 900 feet of work.
>
> The property is equipped with steam, gasoline, water, and air power, both at the mine and the smelter. There are three hoists and an 8-drill air compressor at the mine. An excellent machine shop is situated at the smelter. The smelter has two 125-ton blast furnaces 44 by 160 inches at the tuyeres but is without converters.”

The Empire Copper Company operated almost continuously from 1907 to 1921, shipping crude ores to Salt Lake smelters. In October 1921, the Idaho Copper Company succeeded the Empire Copper Company and installed a mill and tramway. Milling began in 1924, and both concentrates and crude ores were shipped to Salt Lake smelters until operations ceased in 1930. From 1928 to 1930, the mine was worked by Mackay Metals, Inc. which went into voluntary receivership in 1931, at which time the patented claims were taken over by Custer County. A small amount of crude ore was produced by lessees in 1935-1937. The Mackay Exploration Company took over the property in 1939 under lease and bond agreements with Custer County and with Mackay Metals, Inc. (Farwell and Full, 1944).

In 1942, USBM mapped, drilled and sampled the Empire Mine concurrently with the US Geological Survey. Twenty-one underground core holes, and nearly 400 samples were taken, which included samples from level 300 under the oxide mineralization identified on surface. The mine was surveyed and mapped on at least eight of the nine main levels and a small mineral resource was delineated for sulfide mineralization in an orebody at the northern end of the level 1000 (Farwell and Full, 1944).

During the USBM's 1942 survey, mineralization was accessible on at least six of the nine primary production levels and was reported by the USBM to be hosted in overall good ground (geotechnical) condition. In all, the USBM estimated that over 60,000 ft (18,300m) of development were made into the Empire orebody, although only 35,000 ft (10,650m) were accessible at the time. Historically, the main mining method used in the Empire Mine was shrinkage stoping.

Various companies carried out drilling exploration in the AP Pit area between 1964 and 1972. These companies include the Cleveland Cliffs Iron Co. (CCDH 2-9, 1962), New Idria Mines (NI 1-20, 1967), Hile Exploration Co. (H 1-58, 1969), Capital Wire & Cable Co. (CW 1- 14, 1970), and US Silver and Mining Corp. (Behre Bolbear: BDH 1-41, 1972). All holes were assayed for copper, and nine were assayed for gold (NI series). During 1972 a mill was constructed, and the ore developed by the USBM was exploited at the 1100 level. In 1975, Exxon Company explored for copper and molybdenum. Exxon drilled ten holes that were also assayed for gold. By 1975, a total of 151 holes had been drilled on the property, almost all in the AP pit area. Historic ownership/operation of the Empire Mine through 1977 is summarized in Table 6-1 (Mitchell, 1997).

Table 6-1: Historic Operators of the Empire Mine (Mitchell, 1997) (Reproduced)

Company Name	Officer	Date Incorporated	Charter Forfeited	Year(s) at Mine
White Knob Copper Co.	John W. Mackay, President	[1]	[1]	[1]
MacBeth Lease, Inc.	Ravenal Macbeth	[1]	[1]	1904-1907
White Knob Copper and Development Co	[1]	[1]	[1]	?-1905
Empire Copper Co.	Frank M. Leland, President	June 28, 1907	December 1, 1921 (Company reorganized as Idaho Metals Co.)	1907-1921
Idaho Metals Co	L.R. Eccles, President	October 8, 1921	[1]	1921-1928
(In Receivership)	-	-	-	1928
Mackay Metals, Inc.	W.E. Narkaus, Manager	June 4, 1928	December 1, 1930	1928-1931
(In Receivership)	J. Ray Weber, Receiver	-	-	1931-1936
Mackay Exploration Co.	Ted Cherry, President; J. Ray Weber, Manager	August 21, 1939; Reinstated March 25, 1974	1971; 1974	1939-1960
Custer Copper Corp. (lessee)	W.P. Barton, President	June 28, 1946	Active through 1967	1946-1956?
Idaho Alta Metals Corp. (lessee	E.G. Bowen, Executive Vice President	November 19, 1954; Reinstated January 24, 1957	1956?; November 30, 1959	1956-1958
R.V. Lloyd & Co.	R.V. Lloyd, President	[1]	Company reorganized as Lost River Mines, Inc.	1960-1965
Lost River Mines, Inc. (Empire Copper, Inc.)	R.V. Lloyd, President	March 4, 1965	November 30, 1966	1965-1966
J.R. Simplot Co.	J.R. Simplot, President	February 2, 1946	Active	1970
Honolulu Copper Co.	[1]	[1]	[1]	1972-[1]
Myko, Inc.	Ivan Taylor, Vice President	March 7, 1973	Not reported as active in 1981	1973-1974[1]
Exxon	[1]	[1]	Still Active	Exploration: 1977

The first systematic modern-day exploration was conducted by Cambior Exploration USA Inc. (“Cambior”), who explored the property from 1995 to 1997. This exploration entailed data compilation, surface mapping, surface sampling, and ground and airborne magnetic surveys. In 1996 and 1997, the company drilled 47 core holes (totaling roughly 24,100 ft) on approximately 330-ft (100-m) spaced fences along the N-S strike of the deposit, including 21 in the AP Pit area.

Sultana Resources, LLC leased the property in October 1999, but there is little information available regarding exploration or other activities, including transfer of Project ownership, carried out between 1999 and 2004. In December 2004, Trio Gold Corporation (“Trio”) completed a 10-hole, 2,300 ft (700-m), reverse-circulation (“RC”) and PQ-core drill program in the AP Pit area. The salient results of the program are documented by van Angeren (2005). The drilling program consisted of nine 4.5-in diameter RC drillholes (2,200 ft) and one 3.4-in diameter PQ-core drillhole (95 ft). All of Trio's drillholes were vertical.

RC cuttings were sampled at 1.5 m intervals. Two samples, ranging from 900 g to 5440 g, were bagged from each interval, sample size depending on sample recovery. It is not known how these samples were split. A total of 398 RC samples were sent for assay to American Assay Laboratories (AAL) in Reno and 45 duplicate samples were sent to Loring Laboratories Ltd., Calgary. Trio’s PQ-core hole was drilled at the north end of the AP pit for metallurgical purposes. The core and other bulk sample material were sent for testing at Kappes, Cassiday & Associates Inc. (“KCA”) in Reno, Nevada.

Trio’s drill program was successful in improving the thickness of mineralization to at least 220 ft, and in confirming the grades of copper, gold and silver in the AP pit area. Due to the nearly flat-lying nature of the AP Pit oxide skarn, thicknesses are considered reasonably true. Results of the drilling exploration indicate that copper favors exoskarn, whereas gold is more closely associated with limonitic (FeOx) breccias and stockworks.

Based on the results of Trio’s 2004-2005 drilling exploration, a 65-drillhole infill drilling program, along with comprehensive metallurgy, was planned for 2005 and 2006 (van Angeren, 2005). The new drill locations were proposed to test mineralization below existing drilling, test the precious metals content within the known copper orebody, and extend precious metals testing to greater depth.

In 2006, Journey Resources Corporation (“Journey”) drilled 33 of the 65 holes proposed by Trio. All the drillholes were in the AP pit area focusing on oxide mineralization, with the balance planned for 2007. The 33 holes totaled 13,240 ft and consisted of five NQ core and 28 RC, with two of the RC drillholes lost. Summary significant results from this drilling were reported by Anderson (2007). All drillholes were inclined at -45° to the west, and true thicknesses are considered to be approximately 75% of drilled values.

Journey’s drill program was successful in confirming the grades and widespread distribution of copper, gold and silver in the AP pit area, and further confirmed the results of the previous drilling exploration carried out by Trio. In April 2007, Anderson Resource Associates Inc. produced a technical report on the Empire Mine Project for Trio and Journey. This report included a mineral resource and mineral reserve estimate for the oxide portion of the Empire Mine deposit. The planned next phase of exploration, to complete the remaining 32 drillholes of the 65-drillhole schedule, was conducted by Musgrove in 2011.

Musgrove completed 14,265 ft of RC drilling in 24 drillholes in 2011 (van Angeren, 2014). Seventeen holes were drilled in the northern half of the skarn deposit, in the area subject to the most intense historical underground development. The other seven drillholes were in the AP pit area. All drillholes were inclined at -50° to the west. True thicknesses of the mineralized zones are variable and unknown but are thought to be up to 75% of drilled intervals. Highlights of the 2011 campaign completed by Musgrove were reported by van Angeren (2014).

Finally, in 2013, Boxxer Gold Corporation (“Boxxer”) initiated follow-up work on Trio’s 2005 metallurgical testing by extracting four bulk samples from four test pits representing the four different mineralized rock types encountered. The results of the 2013 Boxxer Gold testwork are presented in Section 13.3 of this report. No other work is known to have been completed between 2013 and Konnex’s acquisition of the Project.

6.2 Historical Estimates

The mineral resource estimate described in the following paragraphs pre-dates current NI 43-101 reporting standards and is not classified according to current CIM definition standards. The historic mineral resource estimate described here is not considered contemporary, accurate or reliable, and is included here for historical completeness only. Konnex does not intend to imply that the historical estimate validates, corroborates or otherwise impacts the current mineral resource statement as presented in Section 14 of this report.

In 1997, based on the results of the drilling exploration completed at that time, Cambior reported mineral resources of 27 million Tonnes grading 0.42% copper, mostly within the AP pit area (Schnabel & Lloyd, 1997, and Cambior, 1997). Cambior used geologically representative search distances of 100 feet and 200 feet from contiguous 5-ft core samples on 20x20x20 foot blocks in an inverse distance algorithmic model, with commercially available Techbase software. Most of Cambior's drill fences are 100m to 330m apart, with holes on 60m centers.

While Cambior’s estimate was based on extensive diamond drilling and was calculated by a reputable Canadian mining company using modern, industry accepted standards and practices, the estimate does not conform to current NI 43-101 Standards of Disclosure. Cambior’s terminology, including "drill indicated oxide resource", does not conform to the requirements of Sections 1.3 and 1.4 of NI 43-101, and a qualified person has not done sufficient work to classify the historical estimate according to modern reporting requirements. Konnex is not treating the historical estimate completed by Cambior as current.

6.3 Historic Production

The Empire Mine produced 694,000 tonnes with a recovery of 3.64% Cu, 1.64 g/t Au and 53.8g/t Ag from underground workings during the period 1901 to 1942 (Farwell & Full, 1944). Actual head-grades are unknown, although mine inspector reports indicate that direct-shipments to the smelter averaged 6% Cu (Anonymous, 1911, 1912 and 1923). A further 115,500 tonnes at 2.27% Cu, 1.11 g/t Au and 23.76 g/t Ag were mined intermittently from 1943 to 1971, (USGS Bull 2064-I, 1995). No production records are available for material extracted after 1971.

Historical production from numerous adits and shafts in the Horseshoe claim block, including the former Horseshoe Mine, Blue Bird Mine, and White Knob Mine, was predominantly from skarnified limestone in contact with porphyritic intrusive bodies. Documented production was predominantly high-grade silver, lead, and zinc occurring as galena and sphalerite. The White Knob and Blue Bird Mines produced in excess of 12 million pounds of lead and 300,000 oz of silver prior to WWII with reported average grades of 10 oz per tonne of silver and 17% lead. Grades at the Horseshoe Mine are reported as 20% lead and over 100 oz of silver per tonne.

7. GEOLOGICAL SETTING AND MINERALIZATION

A portion of the text presented in this section is modified and/or excerpted directly from '*An Independent Competent Person's Report on the Empire Mine, Idaho, USA*' prepared by SRK (SRK, 2017).

Idaho's geology is well known and described in many sources; these have been summarized and compiled on the "Digital Geology of Idaho" website, which has been sourced for this report section.

7.1 Regional Geology

The Empire Project is situated within the Idaho-Montana fold and thrust belt, a segment of the vast Cordilleran orogenic belt. This belt was formed over a significant timeframe encompassing both the Sevier Orogeny (~160-70 Ma) and Laramide Orogeny (~70-30 Ma), and it represents an area of overlap between these two major orogenic events. Subsequently, the belt was affected by "Basin and Range" tectonism, a result of tectonic relaxation (~30-10 Ma). The Idaho-Montana fold and thrust belt exhibits distinct characteristics associated with both the Sevier Orogeny and the Laramide Orogeny.

The Sevier Orogeny, driven by steep subduction of the Farallon Plate, resulted in thin-skinned thrusting, accretion of newly formed crust, and emplacement of large granitic intrusions like the Idaho Batholith. These shallow thrust faults primarily affected Paleozoic and Proterozoic sedimentary strata and did not significantly deform the underlying crystalline basement of the Wyoming craton.

In contrast, the Laramide Orogeny, characterized by flatter subduction, produced thick-skinned thrusting, deforming both basement rocks and a thin sedimentary cover. Recent studies suggest that thin-skinned and thick-skinned thrusting may have overlapped temporally or even interacted within the region (Parker & Pearson, 2021). This interaction could be linked to the propagation of Sevier-style thrusting beyond specific geological boundaries, triggering the activation of basement-involved thrusting. Laramide intrusions associated with deeper structures are generally smaller compared to typical subduction-zone batholiths.

The northwest-southeast strike of the thrust faults within the Idaho-Montana belt mirrors the dominant structural orientation of faults observed in the Basin and Range Province. The Basin and Range regional extensional event involved the reactivation of previously shortened regions by normal (extensional) faults. Notably, some Basin and Range normal faults coincide with reactivated thrust faults from earlier orogenic events (Figure 7-1).

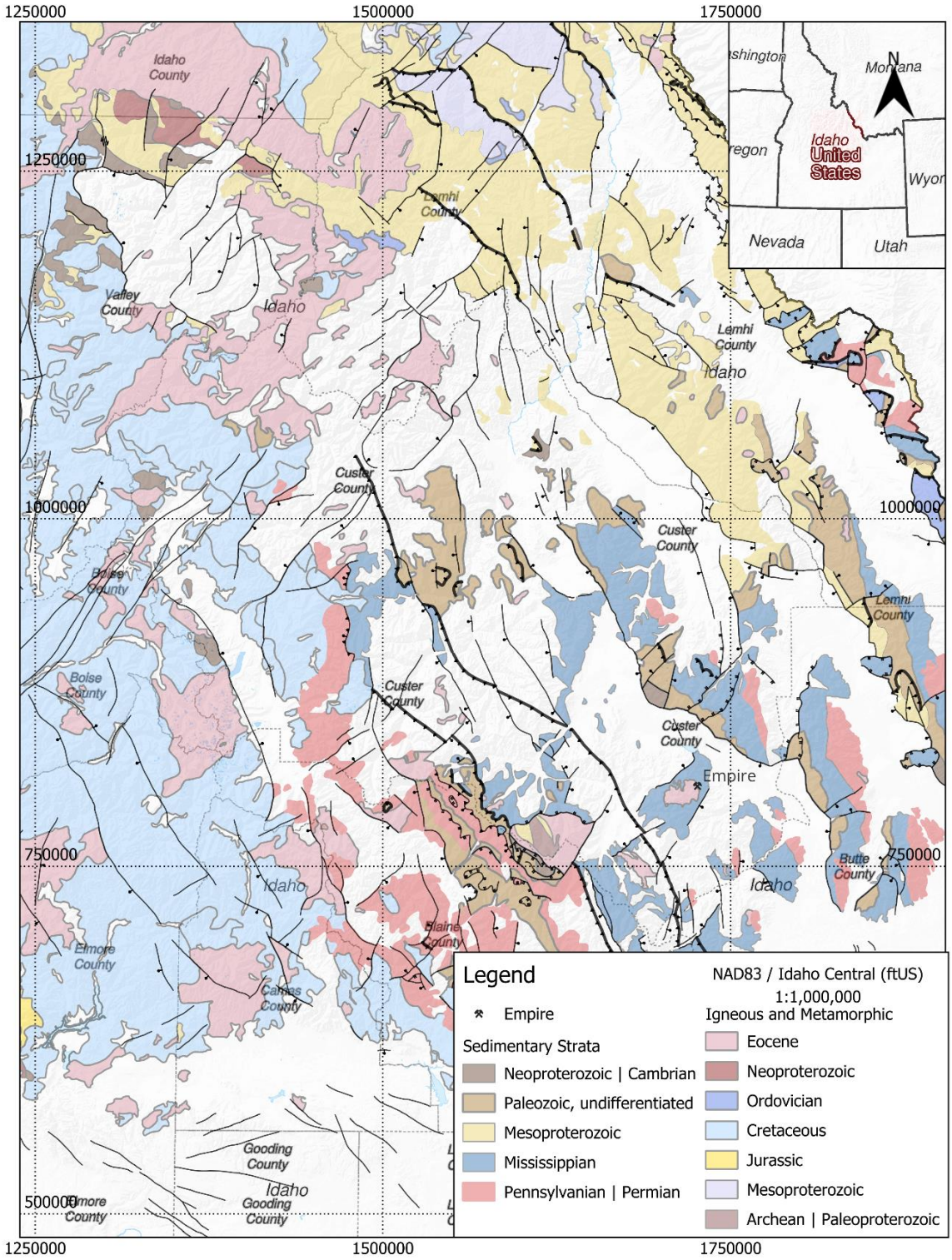


Figure 7-1: A Regional Geologic Map Portraying the Idaho-Montana Fold-Thrust Belt, extending from Central Idaho to Southwestern Montana (Not Shown). (Modified from Parker and Pearson, 2021).

As a product of the Sevier-Laramide orogeny, the North American Cordillera (Figure 7-2) emerged as a long mountain belt approximately 52 million years ago. The continued eastward subduction of remnants of the Farallon-Kula plates led to the intersection of mid-ocean ridges with the subduction zone, resulting in a transition from a subduction zone to a transform plate boundary. Major crustal extension is thought to have begun in Eocene-Oligocene time, which is concurrent with the development of the northern Queen Charlotte transform plate boundary and the Challis magmatism. The Mackay Intrusive Complex intruded (~45-40 Ma) during the transition from Laramide subduction related deformation to a transform plate along the northwest margin of the Wyoming province within the Trans-Montana orogenic zone.

The Trans-Montana orogenic zone (Figure 7-3) developed in Paleoproterozoic time (1.9-1.8 Ga) and coincides in part with the Great Falls Tectonic Zone of O'Neill and Lopez (1985); extending southward from the Great Falls shear zone to the Dillon shear zone. Reworking of the northwest margin of the Archean Wyoming province during collision and subsequent convergence with the accreted Paleoproterozoic Wallace terrane and accompanying Archean micro-continents (Medicine Hat block and the Pend Oreille domain) resulted in the foreland Trans-Montana fold and thrust belt. This belt extends southward from the Dillon Suture Zone to at least the Madison Mylonite Zone (Sims and others, 2004). Reactivation of these foreland structures within the Paleoproterozoic Trans-Montana orogeny in the Late Cretaceous-Eocene provided favorable conditions for the emplacement of granitic plutons and related mineralization (O'Neill and others, 2002), including the subsequent emplacement of the Mackay Intrusive Complex.

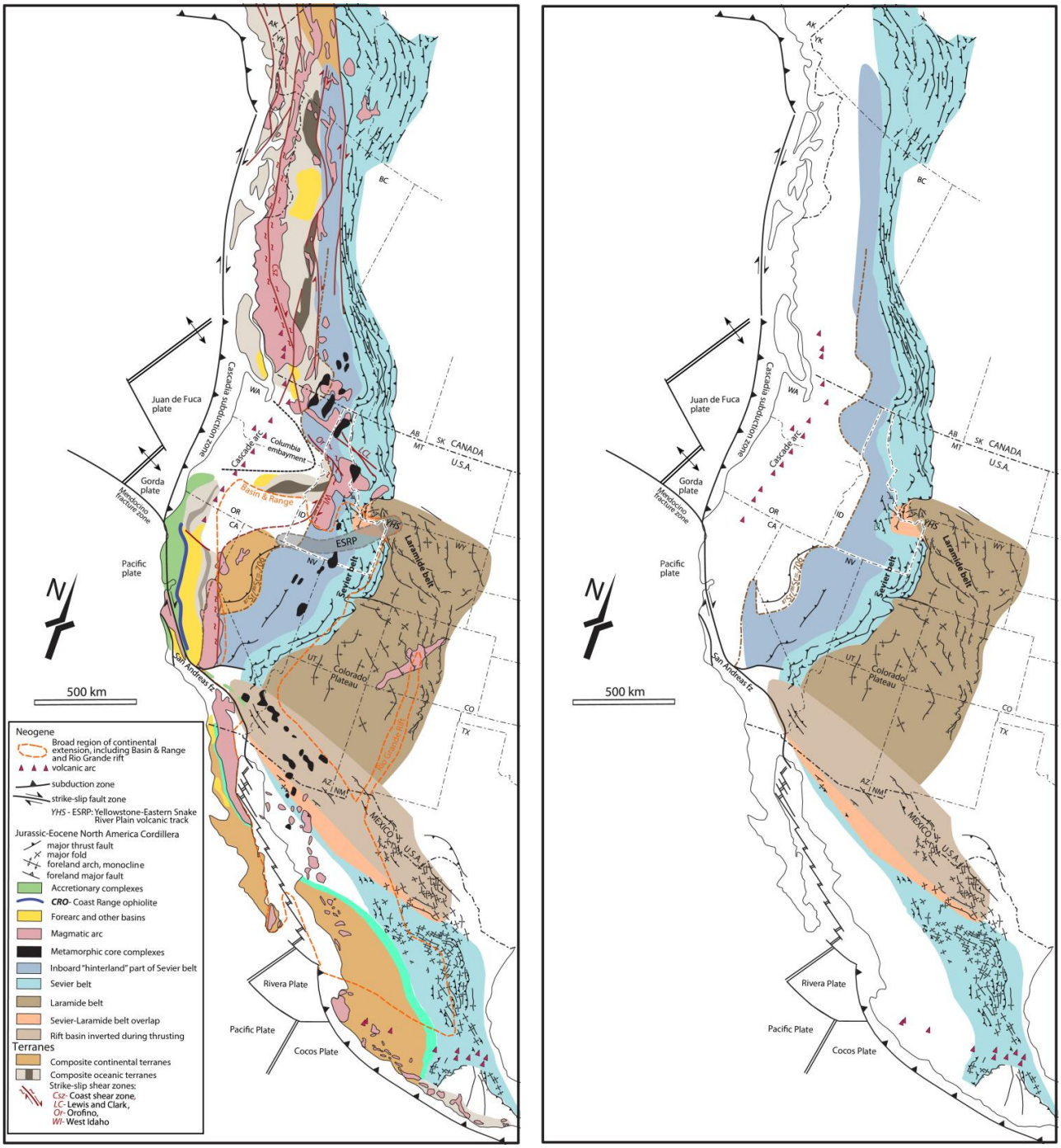


Figure 7-2: Map of the North American Cordillera, illustrating its Major Tectonic Features Adapted from Sources including Yonkee and Weil (2015) and Fitz Diaz et al. (2018)

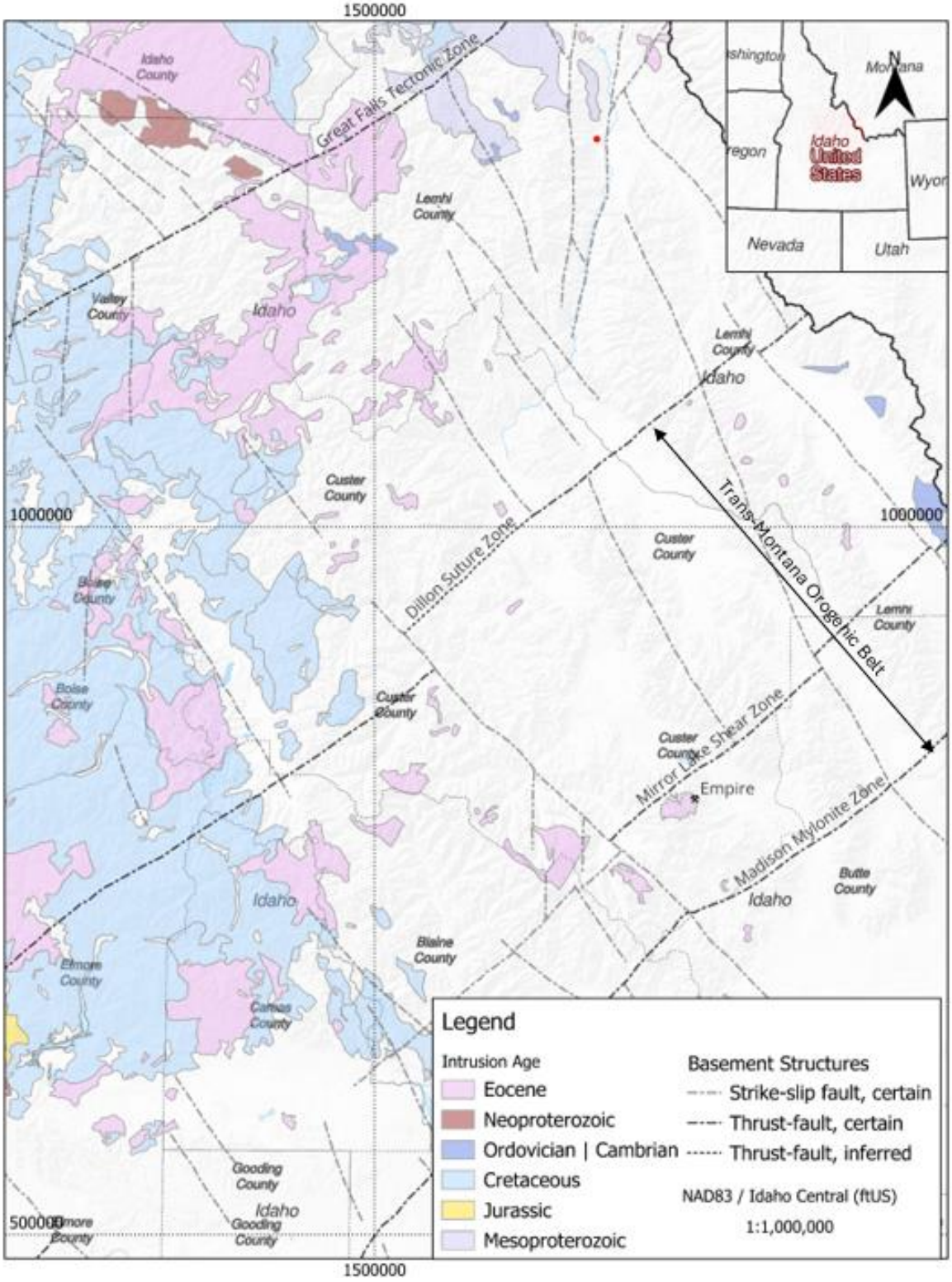


Figure 7-3: Map Adapted from Sims et al. (2005), Focusing on the Distribution of Basement Structures in Idaho and Highlighting Major Intrusive Events

7.2 Local and Property Geology

The Empire Mine Project is located within the Alder Creek Mining District of east central Idaho, positioned east of the Idaho Batholith and north of the Snake River Basalt Plain. This district sits within the North American Cordilleran thrust belt at the northern edge of the Basin and Range structural province.

The geologic framework of the district comprise a thick series of Mississippian Carboniferous limestones intruded by the MIC batholithic mass comprising late Cretaceous granite to early Eocene porphyry, traversed by narrow dikes of trachyte porphyry closely related to the granite porphyry in age and composition but differing markedly in general appearance. Along the eastern and southern margins of the district, erosional depressions are occupied by a thick series of Eocene lavas and tuffs, partially obscured by Quaternary gravels (Figure 7-4).

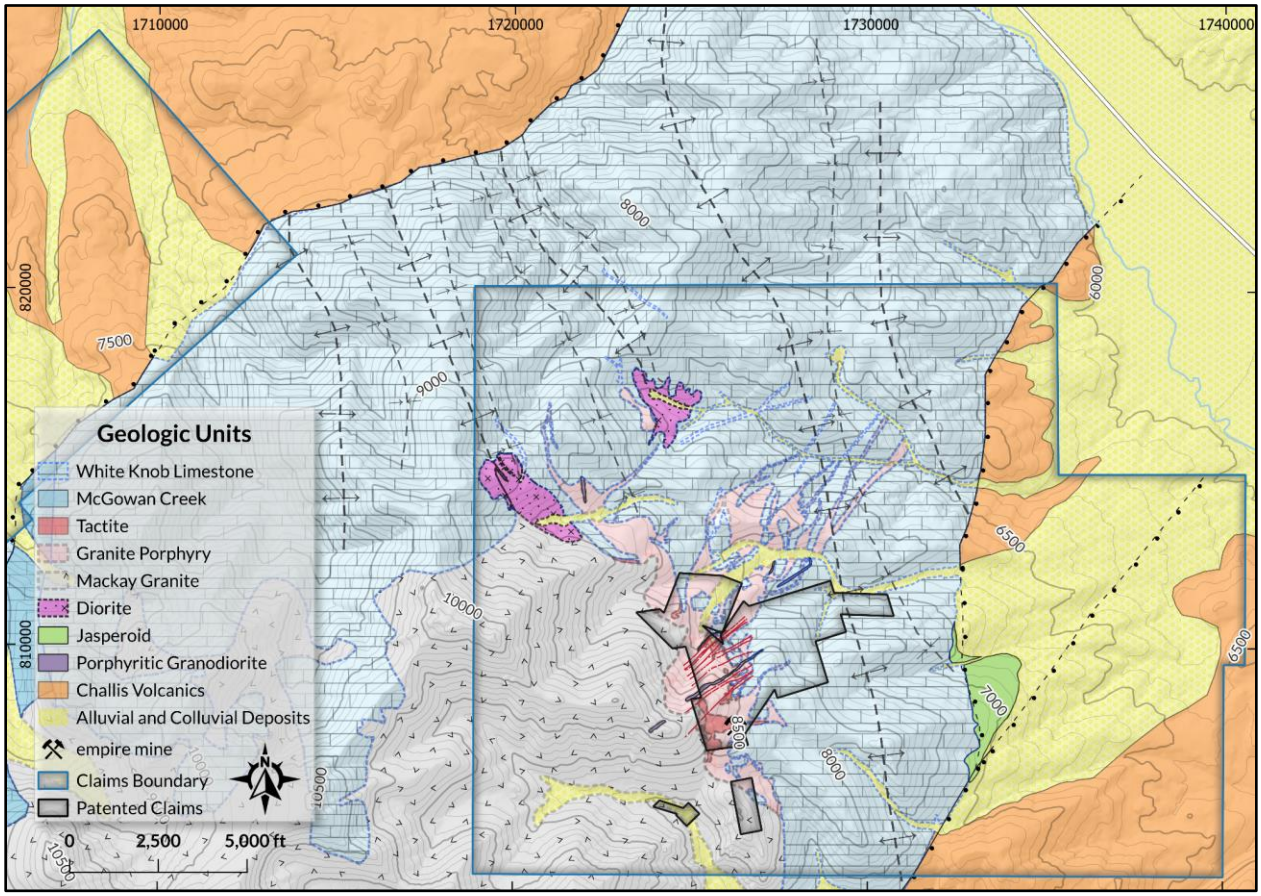


Figure 7-4: Map of the White Knob Horst and Empire Mine (Farwell & Full, 1944; Skipp et al., 2009)

The Empire Project area overlies a north-trending contact zone between an Eocene granitic complex, including the Mackay Granite and Mackay Porphyry, and the Upper Mississippian age White Knob Limestone. This contact zone hosts a garnet-pyroxene-magnetite skarn developed in both the carbonate and intrusive rocks. The skarn is notably significant, serving as the host of the polymetallic copper mineralization characteristic of the Empire Mine. The intrusive contact is sharp and dips steeply eastward.

7.2.1 Lithology

Bedrock geology in the Empire Mine Project area comprises Paleozoic sedimentary rocks, the Mackay Intrusive Complex, and Tertiary volcanics. The oldest rocks in the Project area belong to the McGowan Creek and White Knob Limestone Formations, ranging in age from Early to Late Mississippian. The White Knob Limestone is characterized by medium grey to greyish-black coloration and exhibits variations in bed thickness, ranging from thin to thick and comprised of interbedded layers of variable grain size ranging from medium to coarse pure bioclastic limestone and spiculitic micrite. Within its stratigraphy, medium grey to moderate yellowish-brown lenses of quartzose pebble conglomerate, medium- to coarse-grained sandstone, and siltstone appear. Additionally, sparse interbeds of thin mudstone and chert are present near the base. Large-scale graded bedding and lateral gradations are commonly observed, and the unit displays a steep westward dip.

Conformably underlying the White Knob Limestone is the McGowan Creek Formation, primarily composed of medium grey to greyish-black siliceous, carbonaceous mudstone or argillite. This unit, characterized by medium- to thin-bedded layers, may locally weather to greyish-red-purple hues. Intercalated light grey limestone, fine-grained quartzose sandstone, and blue-grey chert beds, which weather to very light grey are present in the upper part of the section. The sedimentary strata collectively exhibit folding, with anticlinal and synclinal axes generally trending north-northwest. The fold limbs dip moderately to steeply to the northeast and southwest.

The sedimentary rocks are both intruded and overlain by Tertiary igneous rocks. The MIC, covering approximately 7,413 acres, exhibits irregular apophyses extending into the surrounding sedimentary rocks. The sequence from oldest to youngest includes quartz monzodiorite, granophyre, Mackay Porphyry, Mackay Granite, and numerous veins/dykes of latite porphyry, granodiorite porphyry, and quartz porphyry.

The oldest unit, quartz monzodiorite, is characterized by a plagioclase framework hosting interstitial quartz and K-feldspar. Accessory minerals include magnetite, zircon, rutile, and apatite. Notably, this unit exhibits alteration by chlorite and sericite.

Granophyre, distinguished by its extensive quartz-K-feldspar intergrowths in the groundmass, contains sparse phenocrysts of K-feldspar, minor plagioclase, and trace clinopyroxene and amphibole. Accessory minerals include fluorite, magnetite, and trace amounts of zircon, titanite, and rare earth elements.

Mackay Porphyry exhibits a typical porphyritic texture with a fine-grained groundmass. Phenocrysts include K-feldspar (large and euhedral), smoky quartz (rounded and exhibiting a vermicular texture), oligoclase, and minor amphibole and/or biotite. The groundmass composition differs from the phenocrysts, containing less calcium-rich plagioclase and more potassium-rich K-feldspar. Notably, the amphiboles in this unit are calcitic and exhibit variable compositions, while biotite shows high fluorine content.

The Mackay Granite, the major intrusive phase, displays both porphyritic and rapakivi textures with a fine-grained groundmass. Like granite porphyry, the groundmass composition differs from the phenocrysts in terms of plagioclase and K-feldspar content. The presence of ferro-edenite amphibole (high in fluorine) and high-fluorine biotite further distinguishes this unit. Additionally, the Mackay Granite exhibits unidirectional solidification textures and lacks groundmass at contacts with granite porphyry, showing instead coarse-grained feldspars and smoky quartz.

Various granodiorite and aplite dykes intrude all other formations and appear to postdate skarn formation. Aplite also forms a seemingly plug-like mass underlying the site at shallow depth. Aplite does not appear to have caused skarn-formation in the White Knob Limestone.

Skarnification within the Mackay Porphyry was facilitated by its high fluorine content. The presence of fluorine, a volatile, indicates that the porphyry was a “wet” intrusion. This characteristic would have enhanced fluid and mineral transfer between the intrusive and wall rocks, resulting in calc-silicate skarnification rather than simple thermal metamorphism of the limestone. In contrast, the Mackay Granite, characterized as “dry” (depleted in volatiles), did not induce skarnification in the limestone.

Field relationships observed in the mine workings and in core confirm that the Mackay Porphyry was an early, volatile-rich, apophysis of the Mackay Granite, that was subsequently intruded by the granite and its accompanying aplite dykes. Various granodiorite and aplite dykes intrude all formations and appear to postdate skarn formation. Aplite also forms a seemingly plug-like mass underlying the site at shallow depth and is not related to skarn-formation in the White Knob Limestone.

The Empire Mine calc-silicate skarn forms a 500-ft wide sinuous belt extending for over 8,200 ft along the limestone - porphyry contact from the south end of the property to the White Knob Mine (Figure 7-5). The skarn consists of garnet with significant quantities of diopside, along with subordinate amounts of magnetite, haematite, actinolite, scapolite, wollastonite, epidote, and fluorite. Three identified skarn types, listed in order of decreasing abundance, include:

I. Green Exoskarn (derived from limestone) Primarily composed of well-banded diopside-garnet ± laminae of coarse magnetite grains.
II. Brown Endoskarn (derived from porphyry) Massive, sucrosic and garnet dominant. Most of the garnet is iron-rich brown andradite and translucent-yellow grossularite (Umpleby, 1917).
III. Magnetite skarn occurs as massive, fine-grained, crudely bedded magnetite with rare “rip-up” clasts of exoskarn, and less commonly, as magnetite-cemented breccia with abundant fragments of exoskarn and/or endoskarn.

All three skarn-types contain subordinate amounts of hematite, actinolite, scapolite, wollastonite, epidote, and fluorite. Large exoskarn masses appear to have rafted into the porphyry as pendants and embayments. The inner edges of the exoskarn bodies often grade to massive magnetite skarn at their contact with endoskarn or porphyry, particularly evident at the southern edge of the large exoskarn mass underlying the Atlantic-Pacific Open Pit. Remnant bedding can be traced into the magnetite from the rest of the exoskarn body. At the outer margin of the skarn, a narrow, discontinuous belt of marble separates it from fresh limestone. Magnetite breccias may represent the pathways providing access of mineralizing hydrothermal fluids into the then developing skarn.

The Empire Mine skarn is cut lengthwise by several linear bodies of gossanous, clay-altered, iron oxide breccia (FeOx breccia), which may represent post-skarn faults. These structures host copper-gold mineralization.

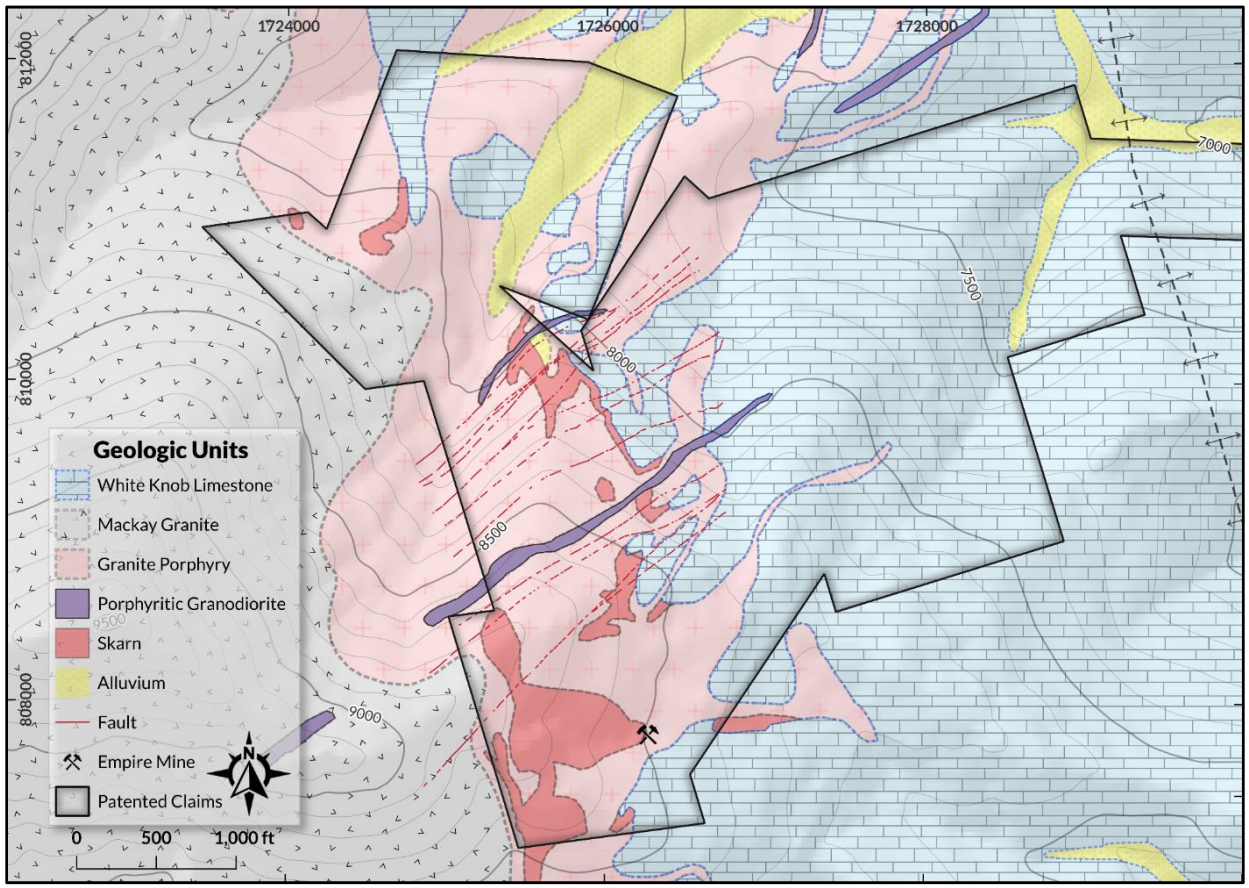


Figure 7-5: Map of the Empire Mine (Farwell & Full, 1944)

7.2.2 Structure

The Project area occupies a portion of the Idaho-Montana fold and thrust belt, and specifically the White Knob thrust plate, which is bounded by the Cretaceous Copper Basin Thrust to the southwest and the Big Lost River Thrust to the northeast (Figure 7-1). Within the White Knob thrust plate, two northeast-striking Eocene faults define the northwest and southeast margins of the White Knob horst. Within the White Knob horst, Mississippian sedimentary rocks are folded with anticlines and synclines generally trending north-northwest, with some local variation. Fold limbs dip moderately to steeply to the northeast and southwest (Figure 7-4). Previously, the uplift of the horst and emplacement of the pluton were thought to be synchronous. However, recent studies propose a potential link between the uplift and the Eocene-Oligocene crustal extension associated with Challis magmatism.

In addition to the dominant northeast-striking extensional structures including the horst, faults, intrusions, and dyke swarms, there are also northwest-striking Neogene faults in the vicinity of the Project area. These faults are a product of Basin and Range extension and are prevalent throughout the Challis Volcanic Group within and on the northwest and southeast sides of the horst (Figure 7-4).

The Laramide orogeny is considered an oblique subduction with transpressional regional strain and faulting partitioned into arc-parallel strike-slip faults and arc-transverse faults. Volcanism and the deposition of intermediate- and high-sulfidation deposits are associated with structural features such as flexures, pull-apart basins, and jogs or stepovers in strike-slip faults. Dikes, veins, and mineralized normal faults exhibit

arc-oblique to orthogonal orientations, often extending locally along bounding strike-slip faults. Extensional grabens in the back-arc may also be aligned oblique to the arc (Rhys et al., 2020).

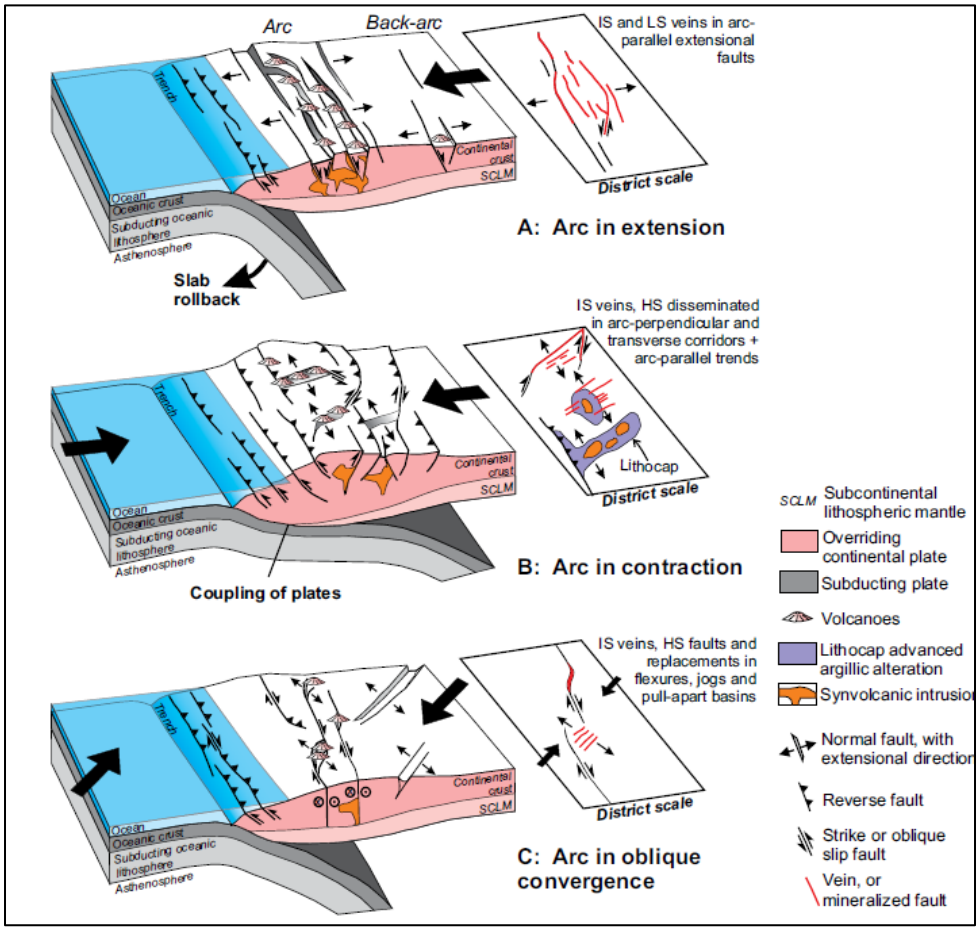


Figure 7-6: Schematic Illustration of Various Arc Stress States and Tectonomagmatic Settings of Convergent Margins (Rhys et al., 2020)

Evidence of this structural regime is evident in the dike swarm cutting through the Mackay Intrusive Complex and in the shear zones mapped in the underground workings. The N35E orientation of these second-order structures facilitated the emplacement of porphyry dikes. These dikes are observed to intrude along these zones of weakness, suggesting a genetic relationship.

The dikes and shear zones acted as conduits for the ascent of mineral-rich hydrothermal fluids derived from the underlying cooling magma. The enhanced permeability associated with these zones facilitated the movement of hydrothermal fluids, which were crucial for driving metasomatic alteration of the surrounding rock units.

The contact zones between the porphyry dikes and the surrounding limestone units became preferential locations for metasomatism. This is attributed to the chemical differences between the rock types. The interaction between the fluorine-rich hydrothermal fluids and the chemically reactive host rocks resulted in intense fluid-rock interaction, leading to the development of skarn along these contact zones.

7.3 Alteration and Mineralization

Mineralization at the Empire Mine primarily manifests as a skarn-hosted polymetallic system (Cu, Au, Zn, Ag), and is accompanied by a subsequent epithermal event along pre-existing structures, present as overprinting iron oxide breccias resulting in gold and silver mineralization (Au, Ag, ±Cu). Historical results and mining records suggest that skarn mineralization at Empire may exhibit depth zonation with copper giving way to zinc and finally tungsten mineralization. The exact process of this zonation is as yet unknown.

7.3.1 Skarn Mineralization

Copper-oxide (carbonates, malachite, and azurite) and sulfide (chalcopyrite/chalcocite) mineralization occur to varying degrees within exoskarn in rafted limestone fragments and endoskarn in porphyry. The mineralization occurs as copper oxide veinlets and stockworks and disseminated oxides and sulfides. The sulfides have similar characteristics, but also occur as massive lenses (both copper sulfides and magnetite) along skarn-hosted fault breccias. The degree of mineralization in skarn-hosted fault breccias is a function of the amount of contained skarn fragments. Copper and iron were introduced into the skarn during the latter stages of the scarification processes (Chang, 2003). Brittle faulting/shearing and ductile deformation during the scarification process likely provided the conduits for mineralizing fluids. Magnetite breccias may represent the pathways providing access of mineralizing hydrothermal fluids into the then developing skarn.

At the northern end of the property, mineralized zones dip eastward between 45° and 90°, somewhat parallel to the limestone-porphyry contact (but cross-cutting the west-dipping limestone). At the southern end, in the vicinity of the AP Pit area, the dip of both exoskarn and mineralization ranges from 30° to 50° towards the east.

Drilling has identified a skarn-hosted body of disseminated and stockwork copper-oxide mineralization extending over a strike length of 4,000 ft, with a thickness of 6 to 240 ft from surface, and a width of up to 420 ft. The “width” figure is a function of topography; the skarn is exposed along a steeply-inclined north-trending ridge-crest, with the northernmost outcrop being 840 ft lower in elevation than the southernmost exposure. All of the mineralized intercepts are in endoskarn, exoskarn and skarn-hosted breccias. The mineralization intersected is oxidized from surface to a vertical depth of approximately 400 ft, with sulfide mineralization dominating below that depth. The transition zone between oxide and sulfide extends over tens of feet.

7.3.2 Iron Oxide Breccia Mineralization

The Empire Mine skarn is overprinted by a series of north-trending anastomosing faults which are represented by gossanous breccias, veins and stockworks up to several feet in width. Herein termed “FeOx breccias”, these structures consist of intensely clay-altered, chalky and brecciated wallrock (exoskarn, endoskarn and porphyry) cemented by siliceous limonite and goethite (sulfide derived iron-oxide?). Brecciation clearly post-dates scarification. The breccias exhibit strong epithermal signatures, indicated by advanced argillic alteration (clay+pyrite+silica), and open-space textures. These FeOx breccias are gold-bearing and represent a late stage, epithermal, gold-rich, hydrothermal regime overprinted upon the skarn. The copper in these epithermal structures may have been scavenged, in-part, from the pre-existing skarn.

The highest-grade mineralization at the Empire Mine occurs as a poorly defined, steeply dipping, locally iron-rich, 15- to 50-ft thick, copper-gold zone located within and below the large body of skarn-hosted disseminated copper mineralization. Drill core indicates that the skarn in this high-grade zone has been sheared, brecciated, and overprinted with iron oxides (FeOx brecciation). This structure may have been active throughout skarn formation and may have been the major pathway for both the skarn-aged copper

mineralization and the late-stage gold mineralization. In the deeper levels of the mine, this structure contains lenses and veins of copper-bearing massive sulfide. This higher-grade zone forms the bulk of the historical Empire Mine, which has been partially worked for 1,150 ft vertically and 3,000 ft laterally. The near-surface oxide mineralization is interpreted to remain open along strike. The higher-grade sulfide zone, which underlies the oxide zone, is open in all directions, and remains virtually unexplored.

The scale of the processes of skarn development and associated hydrothermal mineralization is characteristic of a large (2 mile long by 130-500 ft wide) skarn system flanking a poorly understood parent intrusive body measuring 1,150 ft in the north-south extent and some 820 to 1,640 ft in width (Maund, 2016). Previous exploration at the Empire Mine has primarily focused on a shallow copper oxide resource comprising a 1,300-ft section of the 2-mile length of the skarn body and has largely discounted or ignored supergene and sulfide Cu, Au, Ag, Zn, W mineralization.

8. DEPOSIT TYPES

A portion of the text presented in this section is modified and/or excerpted directly from '*An Independent Competent Person's Report on the Empire Mine, Idaho, USA*' (SRK, 2017). The author has reviewed this information and the available supporting documentation in detail, and finds the descriptions and interpretations presented herein to be reasonable and suitable for use in this report.

Mineralization at the Empire Mine is primarily representative of a polymetallic skarn deposit, however, epithermal veining has also been identified, both of which represent a continuum deposit style related to igneous intrusions. Porphyry copper deposits often occur adjacent to and below skarn deposits, and although no porphyry copper system has yet been identified at the Empire Mine Project, several characteristic high grade Pb/Zn + Ag veins occur distal to the skarn in the White Knob Limestone.

Several similar deposits occur in the near vicinity of the Empire Mine, including the White Knob Mine and Copper Basin Mine, located ¾ mi north and 10 miles southwest, respectively, of the Project area. The Copper Basin Mine hosts flat dipping beds of soft, black lime shale uniformly mineralized with a dissemination of pyrite and chalcopyrite (Mitchell, 1999). In the Mackay district, ore bodies have historically been mined along a north-south arc-shaped belt at least three miles long and 1,000 feet wide, with the Empire mine at the center of the belt where pipe-like copper-bearing ore bodies were mined. Lead-zinc bodies have been developed at various mines to the north and south. The distribution of the lead-zinc deposits on either side of the major copper deposits suggests zoning (Farwell and Full, 1944).

Polymetallic skarn systems are known to host mineralization containing a variety of metals including Au, Cu, Pb, Zn, Fe, Mo, W, Ag and Sn. The term "skarn" describes the metasomatic replacement of carbonate rocks by calc-silicate mineral assemblages during contact or regional metamorphism. Mineral deposits associated with skarn assemblages are referred to as skarn deposits and are typically the product of contact metamorphism and metasomatism associated with the intrusion of granite or porphyritic systems into carbonate sediments. The different metals found in skarn deposits are a product of differing compositions, oxidation state and metallogenic affinity of the intrusion that provides the source fluids.

Figure 8-1 illustrates a typical environment in which polymetallic skarns form. When a granite or porphyry stockwork intrudes into a carbonate sedimentary sequence, the fluids associated with the intrusion pass through the contact sediments. This creates prograde hydrothermal alteration of varying intensities as a function of the host sediment composition and reactivity of this with the fluids. Zonation is often characteristically evident in both the alteration suite and tenor of mineralization, both increasing toward the center of the intrusive stockwork.

Skarns can be physically and geochemically related to epithermal vein systems. Epithermal gold systems are metalliferous sources that can host mineralization containing several metals including gold (Au), lead (Pb), zinc (Zn), silver (Ag), mercury (Hg), antimony (Sb), copper (Cu), selenium (Se) and bismuth (Bi). These systems generally form at depths less than 5000 ft. They occur associated with extrusive or near surface intrusive rocks and often occupy normal fault or joint systems within 550 ft of paleotopographic surface. The zones can be formed of simple veins with some irregular development of mineralization chambers, commonly in pipes or stockworks.

Figure 8-1 illustrates a typical environment in which low and high sulphidation epithermal gold deposits might occur. For high sulphidation veins, acidic fluids from the intrusive porphyry system follow a direct structural discontinuity to surface, creating advanced argillic alteration in the wall rock and forming veins, veinlets or breccias. These are generally gold and copper bearing at temperatures of approximately <200°C.

For low sulphidation veins, acidic fluids from the intrusive porphyry system follow a structural discontinuity where they interact with meteoric waters and are neutralized. These neutral fluids then continue to surface creating adularia and sericite alteration of the wall rock before forming veins and veinlets that are gold and silver bearing at temperatures of 100-200°C. Some examples of these gold systems can be observed in the USA at the Cripple Creek deposit in Colorado and the Comstock deposit in Nevada.

The metals observed at the Empire Mine Project indicate the intrusion of an intermediate I-type granite source. The skarn hosted in the White Knob limestone is referred to as a calcic exo-skarn as the metasomatic assemblage is external to the Eocene intrusive.

At the Empire Mine Project, intrusion of the Mackay Intrusive Complex is the fluid and heat source for alteration and mineralization of both the endoskarn and the exoskarn. The reactive, porous, Mississippian age limestone sequence hosts the exoskarn mineralization and is pervasively altered, zoning from hematite-magnetite and epidote-garnet-magnetite close to the intrusions center, to more distal epidote-garnet. Mineralization also exhibits zonation with copper dominant mineralization being replaced distally by zinc and then by tungsten and tin.

The gold and silver mineralization observed at the Empire Mine project have previously been interpreted as later stage mineralization and may cut across and overprint earlier skarn mineralization, classified as late stage epithermal mineralization associated with continued hydrothermal circulation through the Mackay Porphyry.

Both skarn and epithermal deposits are continuum of deposit styles related to igneous intrusions. Other related deposit types include porphyry copper deposits, which commonly occur adjacent to and below skarn deposits, as do iron-oxide-copper-gold deposits (IOCG). Vein-type copper-lead-zinc deposits are often found in the calcareous formations distal to the skarn mineralization. No porphyry copper or IOCG system has yet been detected at the Empire Mine Project.

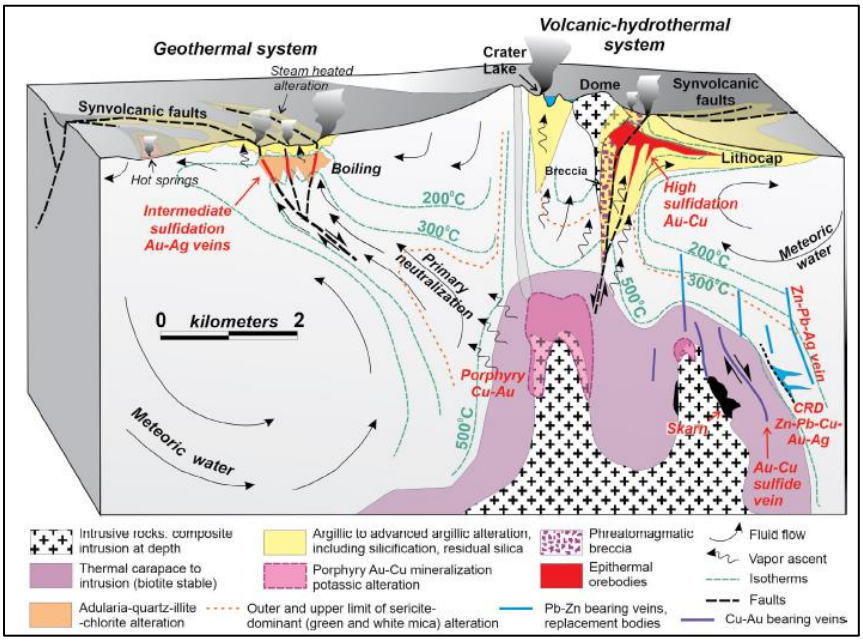


Figure 8-1: Schematic illustration of the Geological Setting and Characteristics of Low- and High-Sulfidation Epithermal Au-Cu Deposits and Sub-Volcanic Porphyry-Type Cu-Au Deposits. (Rhys et al., 2020).

9. EXPLORATION

In 2018, Konnex geologists collected channel samples from rock outcrops in areas difficult to drill due to topographical constraints (Figure 9-1). The channel samples were collected as continuous 5-foot samples, end-to-end, and oriented cross dip as much as possible to mimic the trace of a drill hole and represent the true thickness of mineralization. Konnex geologists attempted to collect sample weights consistent with that of one-half of an HQ core sample. Channel samples were assayed using the same methodology as core and RC chip samples. HRC knows of no other sampling or recovery factors that might materially impact the accuracy of the channel sampling results.

Results of the channel sampling program extended known mineralization to the surface within the proposed pit limit and led to the discovery of the Red Star resource area, which is located roughly 1,000 ft to the northwest of the proposed pit. Of the 329 channel samples collected, 161 contained greater than 0.1% Cu. In the Red Star area, one continuous channel comprised of 12 individual 5-ft samples returned a composite assay of 0.65% Cu, 0.12% Zn, 0.36 g/t Au, and 16.4 g/t Ag.

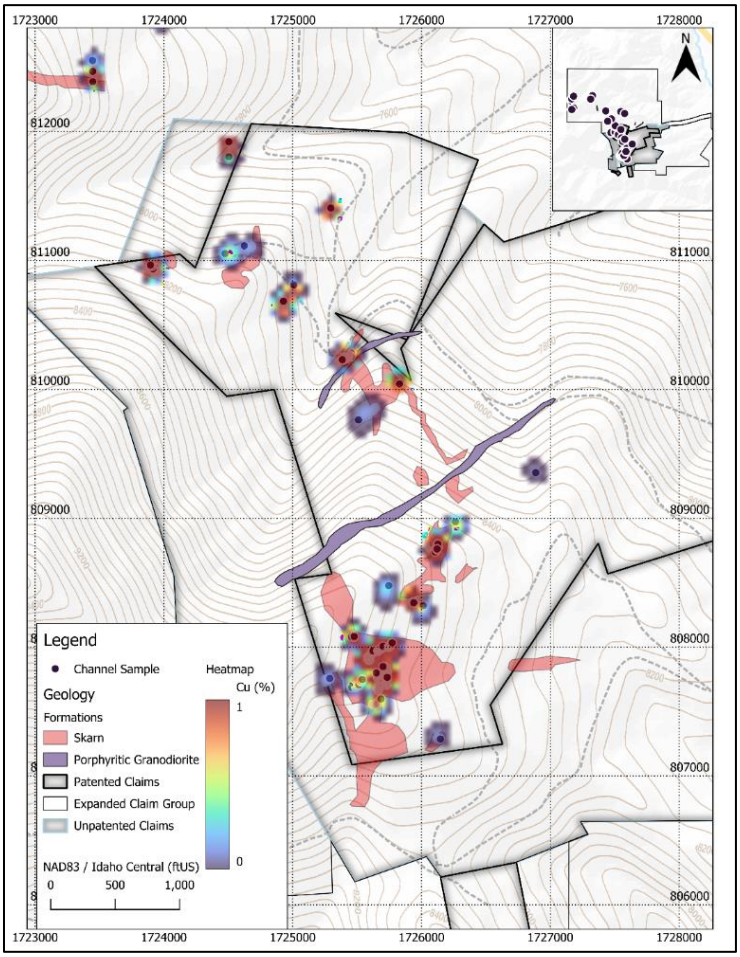


Figure 9-1: Copper Concentration in Channel Samples, Displayed as a Heat Map

HRC is not aware of any other exploration activity, other than drilling, with sufficient supporting documentation or detail to warrant presentation in this report.

10. DRILLING

10.1 Type and Extent

Konnex has completed extensive drilling programs at the Empire Project, totaling 58,592 feet. The programs utilized two primary drill rig types:

- Core Drilling (56 holes): These holes were drilled using a track-mounted drill rig to obtain core samples for detailed analysis. Drillhole core diameter consists of a mixture of PQ and HQ.
- Reverse Circulation (RC) Drilling (147 holes): These holes were drilled with a wheel-mounted prospector buggy rig featuring an articulating boom.

AK Drilling, Inc. of Butte, MT, served as the primary drilling contractor. However, in 2021, Midnight Sun Drilling of Whitehorse, YT, completed an additional 7 RC holes using a track-mounted RC/RAB drill rig.

Figure 10-1 illustrates the drill hole collar locations, and Table 10-1 summarizes the drilling campaigns by year, type, and area.

Table 10-1: Summary of Konnex Drillhole Totals by Year, Area, and Type

Year	Area	Core		RC		Total
		Count	Ft	Count	Ft	Ft
2017	Empire	11	3,999.5	20	5,050.0	9,049.5
2017	Red Star	-	-	1	200.0	200.0
2018	Empire	24	6,934.4	50	15,900.0	22,834.4
2018	Red Star	2	394.5	15	4,460.0	4,854.5
2020	Empire	2	356.4	30	5,085.0	5,441.4
2020	Red Star	-	-	13	2,000.0	2,000.0
2021	Empire	7	4,755.5	6	2,200.0	6,955.5
2021	Red Star	-	-	7	2,880.0	2,880.0
2022	Empire	10	3,502.0	-	-	3,502.0
2022	Red Star	-	-	-	875.0	875.0
Total Drilling		**56**	**19,942.3**	**147**	**38,650.0**	**58,592.3**

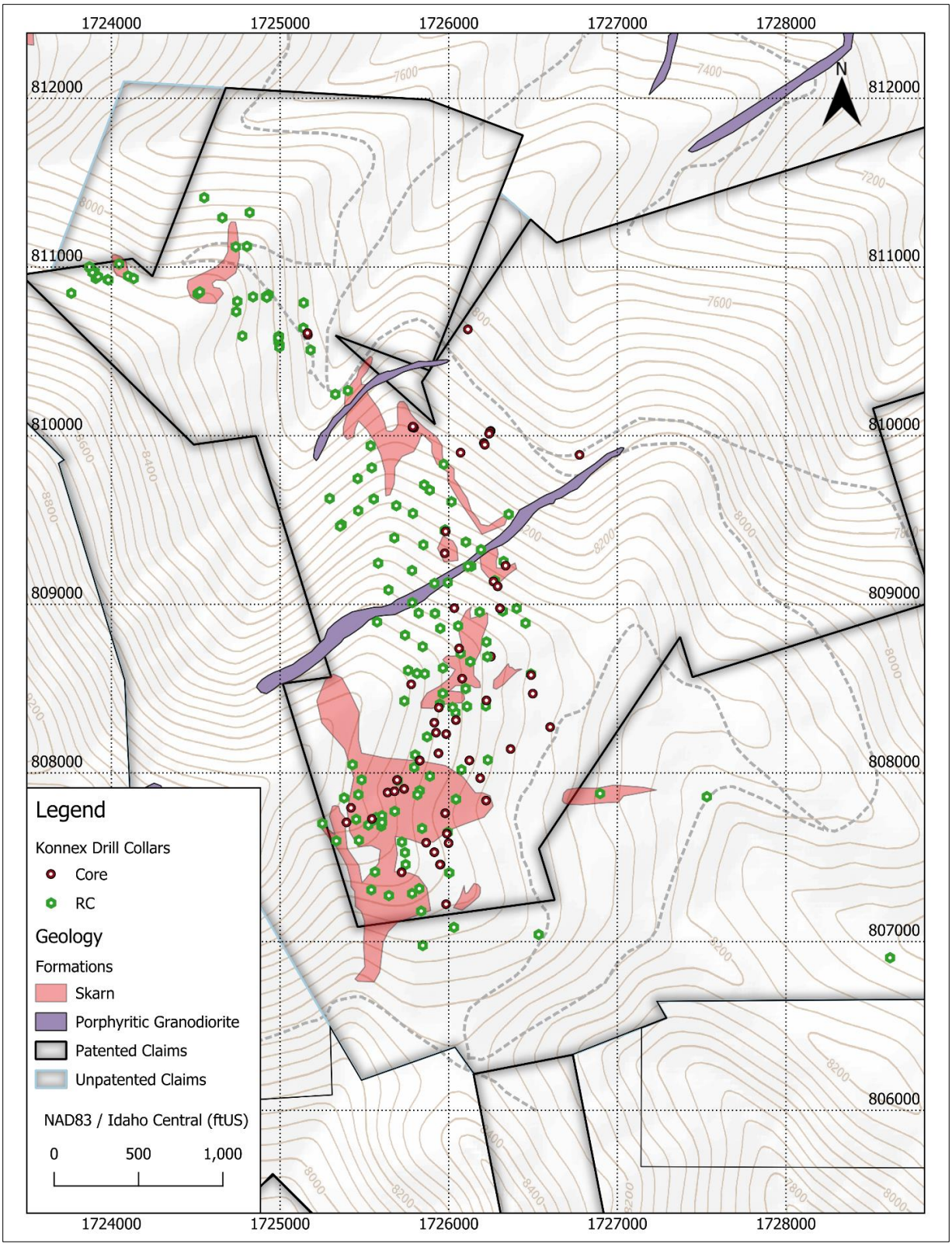


Figure 10-1: Konnex Drillhole Collar Locations

Beginning in 2017, Konnex implemented a multi-phased drilling program to systematically advance the Empire Project. The program encompassed the following key objectives:

- Infill Drilling and Zone Confirmation: Validate and refine historical data by completing infill drilling and confirming the existence and continuity of mineralized zones identified through previous drilling efforts.
- Westward Expansion: Test the lateral extent of the mineralization by drilling westward from the zones delineated by historical drilling.
- Inferred Areas and Resource Evaluation: Target and evaluate the potential of inferred mineralized zones within the proposed pit boundary, aiming to improve geological understanding and resource estimation.
- Peripheral Exploration: Explore additional mineralized zones outside the pit boundary, focusing on the northern and eastern areas of the project.
- Metallurgical Testing: Acquire sufficient core samples for metallurgical and environmental test work.
- Continued Exploration: Conduct ongoing exploration drilling in peripheral areas to identify potential extensions of the known mineralization, particularly focusing on deep sulfide zones.
- Geotechnical Data Collection: Collect additional core samples for geotechnical testing to assess to develop pit slope stability parameters.

The drillhole spacing varies depending on the specific objective of a given hole, but in general the Konnex drillholes are nominally spaced 150’ x 100’ and closer when considering the location of historical drilling. In all campaigns holes are oriented to intersect the projected mineralized zone as close to perpendicular as possible, however in this system mineralization has developed in largely irregular shapes making it difficult to predict the exact angle of interception. Konnex drilling programs tested the orientation of geologic structures, and despite the irregular shape of the mineralized zones, were able to collect representative data.

10.2 Procedures

Drill road and pads were constructed by Getty Construction using the appropriate equipment. Adjustments to collar locations and drillhole orientation were made by the geologist in field to accommodate the spacing requirements of the drill rig.

The drill was aligned on the pads by pre-marking the drillhole alignment with spray paint on the ground and then fine tuning the alignment using a Brunton compass when the drill was set up. The drill-hole inclinations were set using the inclinometer of the Brunton compass. A Konnex geologist visited the rig twice a day during day shift and periodically during night shift. Konnex geologists were responsible for determining when to terminate each drillhole. Konnex’ staff geologists supervised the set-up, drilling and abandonment of all drill-holes under Idaho Department of Environmental Quality (IDEQ) requirements discussed in the following paragraph.

Down-hole surveys were completed on all holes drilled by Konnex. Down-hole surveys were conducted using industry standard digital tools. All drill-holes were abandoned according to IDEQ requirements, capped with a bentonite plug, and marked with a surveyor stake denoting the drill-hole identification. Collar

locations for all drillholes were surveyed with hand-held GPS units by the geologist at the conclusion of the campaign.

RC drillholes were logged in the field by Konnex geologists. For each five-foot interval drilled, a clean metal sieve was placed in the path of the cyclone's waste stream in order to collect a representative sample to retain in a chip tray. Cursory lithologic information was recorded during drilling, and further detail was added after chip tray samples were examined under a binocular microscope. The chip trays, each marked according to drillhole number and footage interval, were photographed prior to being stored in Konnex' secure storage facility in Mackay.

Samples were collected on 5-ft intervals by placing a numbered sample bag in a five-gallon bucket beneath the cyclone sample stream. The cyclone was adjusted as necessary to obtain the maximum sample size possible (and practical) based on the rate of return. Duplicates were collected simultaneously by splitting the material ejected from the cyclone with two buckets with sample bags inside. Excess water was not retained with the samples, and the associated loss of fine material should be considered when analyzing the assay results. Samples were staged and inventoried at each drillhole pad prior to being moved to Konnex secure storage facility.

10.3 Summary and Interpretation of Results

10.3.1 Infill Drilling and Zone Confirmation

The infill drilling program successfully validated and refined historical data. The tighter drill spacing enabled Konnex to confirm the existence and continuity of the mineralized zones identified by previous operators. This validated the geological model and provided a more detailed picture of the controls on mineralization. Step out drilling confirmed mineralization continuity up dip and to the west. Given the shallow-dipping nature of mineralization, significant intercepts are considered to slightly exaggerate the true thickness of mineralization.

10.3.2 Westward Expansion

The drilling program successfully tested the lateral extent of the mineralization in the AP Pit. This step-out drilling confirmed mineralization continuity up dip and to the west, expanding the overall footprint of the mineralization in the southern area of the project.

10.3.3 Inferred Areas and Resource Evaluation

Drilling targeted inferred mineralized zones within the proposed pit boundary in areas with prior uncertainty. While drilling in the AP Pit area confirmed mineralization, other zones exhibited narrower or more constrained mineralization compared to historical data. This discrepancy could be attributed to the inherent variability of skarn deposits. Historical underground workings hampered targeted drilling, potentially leading to intersections of narrower sections of mineralization adjacent to the target. Historical mine maps, for instance, suggest wider zones, and drilling was often prematurely terminated due to encountering these workings before reaching target depths. This is especially prevalent in the area north of the AP pit.

10.3.4 Peripheral Exploration and Continued Exploration

Exploration drilling wasn't limited to the immediate pit area. The program also ventured into peripheral zones, particularly focusing on areas north and south of the project. This peripheral exploration identified additional zones, Red Star, with mineralization, expanding the overall exploration footprint. Furthermore, the program included ongoing drilling in these peripheral areas with a focus on identifying potential

extensions of the known mineralization down dip, particularly targeting deeper sulfide zones. The 2021 core program intersected one these zones down dip from known mineralization on the 700 – 1100 levels. This continued exploration has the potential to further expand the known mineralized zones and potentially discover entirely new skarn mineralized areas between the 700 and 1600 levels.

10.3.5 Metallurgical Testing and Geotechnical Data Collection

The drilling conducted in 2018, 2021, and 2022 prioritized the collection of core samples to define engineering parameters for processing, environmental analysis, and geotechnical pit slope stability. The 2018, core program drilled 23 core holes with sample intervals selected based on lithologic breaks to maximize material for metallurgical testing. In 2022, Konnex drilled 10 core holes in the AP pit area for additional metallurgical sample material. Additional samples were collected from historical and Konnex core that intersect mineralization within the expected pit limits for the geochemical analysis for environmental permitting. Finally, two oriented core holes were drilled in 2021 to collect geotechnical information to determine the angle of the pit wall to be used in mine design.

10.4 Historical Drilling

A description of the historical drilling is contained in Item 6 of this report. Historical drilling carried out by previous operators of the Empire Project accounts for 256 of the drillholes included in the Project database. The earliest drilling was completed in 1943 by U.S.B.M., with various subsequent drilling campaigns completed prior to Konnex' acquisition of the Project. Drilling has been conducted predominately from the surface, except for drilling completed by U.S.B.M in 1943, which was conducted from the existing underground development. The historic drilling database contains 81 core holes totaling 38,375 feet, and 175 RC holes totaling 42,349 feet. All historic drilling was oriented either vertically or angled to the west to perpendicularly intercept the mineralized skarn, and drilling is largely concentrated in the southern portion of the Project area. Historic drillhole collar locations are shown on Figure 10-2, and historic drilling campaigns at the Empire Project are summarized by operator and year in Table 10-2.

Table 10-2: Summary of Drillhole Totals by Year, Operator, and Drillhole Type for Historic Drilling

Year	Operator	Prefix	Drill Type	Count	Footage
1943	U.S.B.M.	B-	Core	11	1,664.90
1962	Cleveland Cliffs	CCDH-	Core	8	2,369.00
1968	New Idria/US Copper	NI-	RC	20	3,002.30
1969	Capital Wire & Cable	CW-	RC	13	1,416.70
	Hile Explorations	Hole-	RC	41	4,616.20
1972	Behre Dolbear	BDH-	RC	41	5,420.90
1975	Exxon	K-	Core	8	7,854.10
1996	Cambrior Inc.	S0	Core	39	20,444.10
1997		S0	Core	8	3,738.20
2004	Trio Gold Corp.	TDD04-	Core	1	97.10
		TRC04-	RC	9	2,241.00
2006	Journey Resources	JDD	Core	5	1,904.10
		JRC	RC	27	10,860.50
2011	Musgrove Minerals	EM11-	RC	24	14,790.91
		EMD11-	Core	1	303
Total Historic Drilling				**255**	**80,723.01**
Total Historic Core Drilling				**80**	**38,374.50**
Total Historic RC Drilling				**175**	**42,348.51**

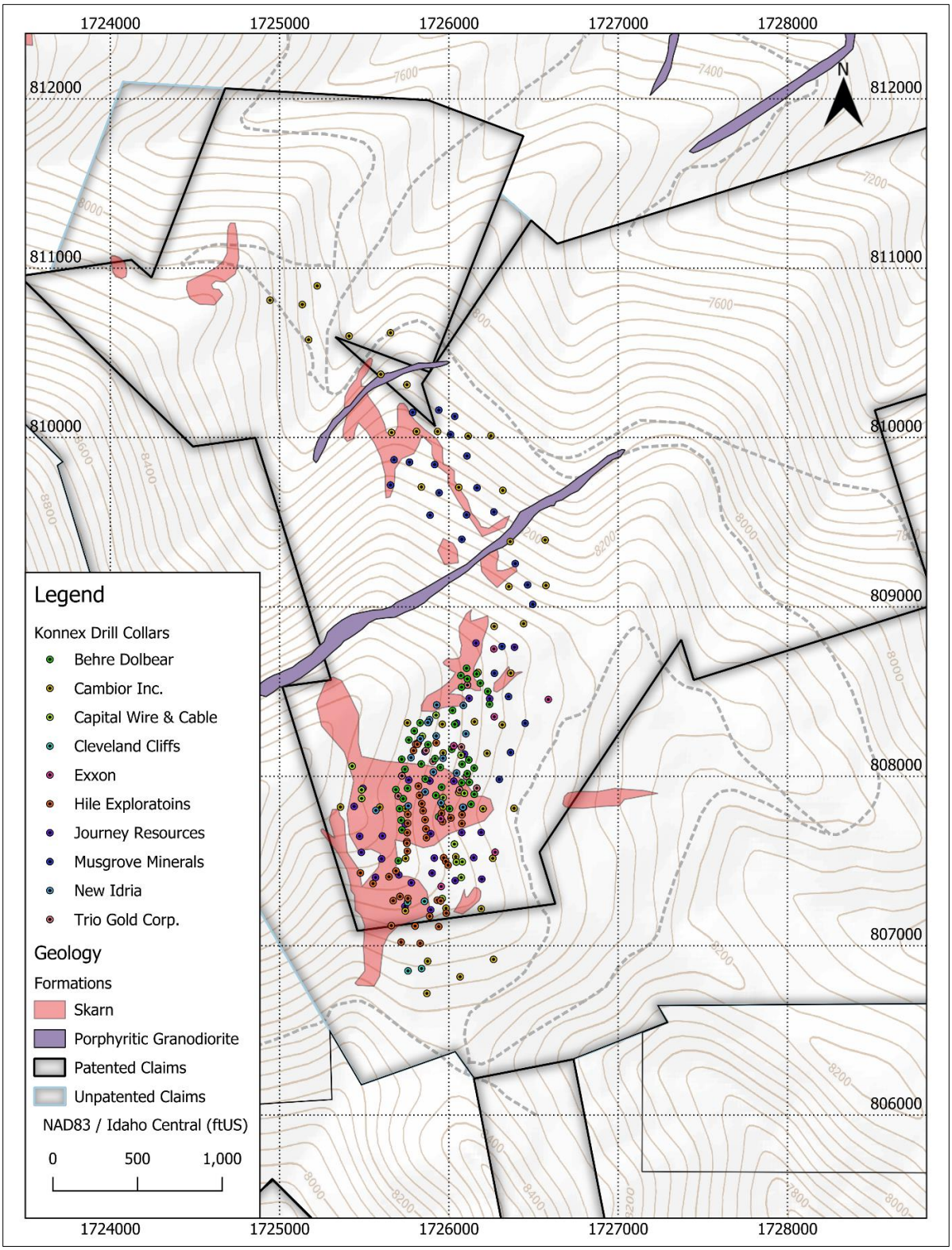


Figure 10-2: Historic Drillhole Collar Locations

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Preparation

Diamond Core: Geologists collected core samples daily, photographing each box and logging recovery, RQD, and geologic data. They then marked the core for sampling at 5-foot intervals, adjusting for lithology and alteration boundaries. Barren limestone sections were excluded. Samples were extended 10 feet above and below mineralized zones. Competent core was cut in half using a tile saw. One half remained in the core box, and the other was placed in a labeled sample bag for analysis. PQ core was quartered, with ¾ retained for metallurgical testing.

RC Drill Cuttings: Samples were collected every 5 feet using a labeled bag placed beneath the cyclone discharge in a bucket. Cyclone settings were adjusted for optimal sample size while maintaining practicality. Duplicate samples were collected concurrently by splitting the material exiting the cyclone into separate buckets with pre-labeled sample bags.

Following sample collection, all core and bagged samples were securely stored on pallets within the core logging facility. To ensure traceability and maintain a chain of custody, each sample bag, for both core and RC drill cuttings, received a unique barcode for identification. A duplicate barcode was attached to the exterior of the bag for verification purposes. After secure storage, samples were transported directly to ALS laboratories by truck transport. Samples remained secured in the back of the truck throughout the trip and were never left unattended overnight. Finally, strict chain of custody protocols were followed, with both ALS and Konnex personnel signing the associated forms for each sample submission.

11.1.1 ALS Methods

All Konnex samples were analyzed by ALS using four acid digestion and induced coupled plasma mass spectroscopy (ME-ICP61) to determine the concentration of 34 elements (refer to Table 11-1 for the complete list). Additionally, all samples were analyzed for gold using fire assay with atomic absorption spectrometry (AAS). Over limits for copper, zinc, lead, and silver were analyzed using four-acid digestion with ICP or AAS finish.

Table 11-1: Suite of Elements Analyzed in ALS ME-ICP61

Element	Limits	Element	Limits	Element	Limits	Element	Limits
Ag	0.5-100	Cr	1-10000	Mo	1-10000	Th	20-10000
Al	0.01-50%	Cu	1-10000	Na	0.01-10%	Ti	0.01-10%
As	10-100000	Fe	0.01-50%	Ni	1-10000	Tl	10-10000
Ba	50-50000	Ga	10-10000	P	10-10000	U	10-10000
Be	5-500	K	0.01-10%	Pb	2-10000	V	1-10000
Bi	10-50000	La	10-10000	S	0.01-10%	W	10-10000
Ca	0.01-50%	Li	10-10000	Sb	5-10000	Zn	2-10000
Cd	0.5-1000	Mg	0.01-50%	Sc	1-10000		
Co	1-10000	Mn	5-100000	Sr	1-10000		

Elements exceeding the pre-defined upper detection limits were subject to further analysis using alternative methods. For copper (Cu), zinc (Zn), lead (Pb), and silver (Ag), samples exceeding ICP detection limits were re-analyzed using either a four-acid digestion with ICP or AAS finish, depending on the specific element. Furthermore, silver and gold samples exceeding their respective overranges underwent fire assay with a gravimetric finish.

Ore-grade copper analysis employed a separate approach. To determine the acid-soluble copper content, samples were subjected to a sulfuric acid leach followed by AAS analysis. Additionally, sulfuric acid/ferric sulfate leach with AAS finish was used for further characterization.

11.1.2 Sample Preparation

The assay procedures have remained consistent throughout the drilling programs (2017-2022) with one exception. The 2017 program utilized the PREP-31 method, which involved splitting coarse rejects using a riffle splitter. For all subsequent drilling programs (2018-2022), the PREP-31Y method was implemented, which employs a rotary splitter for splitting during coarse rejection.

In both preparation methods ALS Minerals crushed the entire sample to 70% passing a 2mm mesh and then split off 250g for pulverization to 85% passing a -75 micron (200 mesh). Cleaner sand was run through the crusher every 5 samples or at any color change in the sample noticed by ALS's lab technicians. Sand was run between every sample in the pulverizing step. The pulps were split again to separate a 30g sample for FA/AAS for gold and a 5g sample for multi-acid digestion ICP-AES for silver and multi-element analysis. Further splits were taken from the same pulp if overruns were required.

11.2 Quality Control/Quality Assurance

QA/QC samples used by Konnex include blanks, standards, and duplicates. Konnex inserts QA/QC samples into the stream at the following frequencies.

- One blank approximately every 10 samples placed randomly in the sample stream and within apparently mineralized intercepts (2 samples per 100);
- One standard every 20 samples; and
- One duplicate approximately every 10 samples placed randomly in the sample stream.

Blank material was sourced from Greensmix® Marble Chips, which is purposed for lawn decoration. The blank material was not certified, but the material was assayed five times to ensure it was inert.

From 2017 to 2022, Konnex progressively incorporated certified reference materials (CRMs) into their drilling program. The program began in 2017 with two commercially available CRMs, followed by the addition of one standard in 2018 and two more in 2022. All CRMs were consistently sourced from MEG Labs in Reno, Nevada.

Core duplicate samples are created at the laboratory during the secondary crushing stage. This involves splitting the original sample when at least 70% of the material passes a #10 sieve. A riffle or rotary splitter is used for this process. RC drill cuttings duplicate samples were collected concurrently at the drill site.

11.2.1 Blanks

Ideally, coarse blanks should achieve a success rate exceeding 80%, with individual blank values at or below three times the detection limit. This necessitates a minimum of four control samples per decision point, translating to one coarse blank analyzed every other batch (eight batches total). Alternatively, a one-in-five failure rate is considered acceptable for coarse blanks.

An evaluation of coarse blank results (Figure 11-1 to Figure 11-3) revealed positive outcomes across all elements of interest. Copper achieved a success rate of 85.8%, exceeding the established 80% threshold.

Silver (97.5%) and gold (96.1%) also performed well, demonstrating minimal background contamination between samples.

It is important to note that the data analysis required the removal of several results attributed to standards prior to completion. This highlights the importance of a robust QA/QC program to identify and address potential issues within the analytical data.

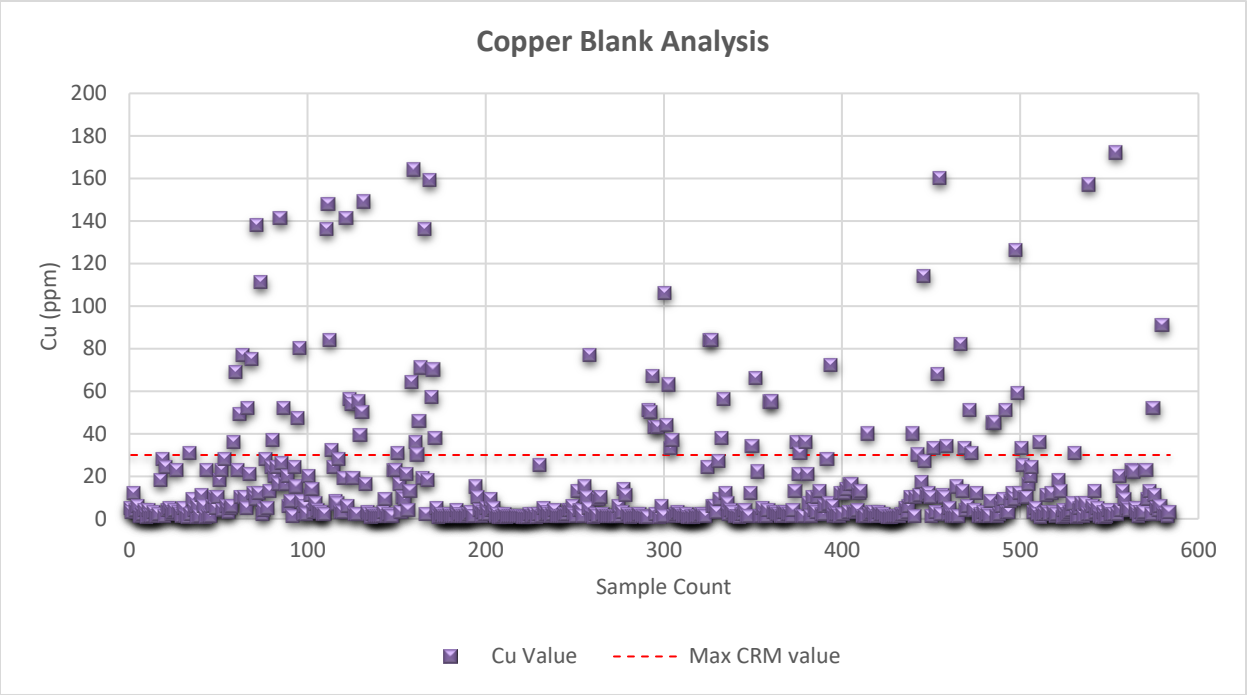


Figure 11-1: Copper Blank Analysis

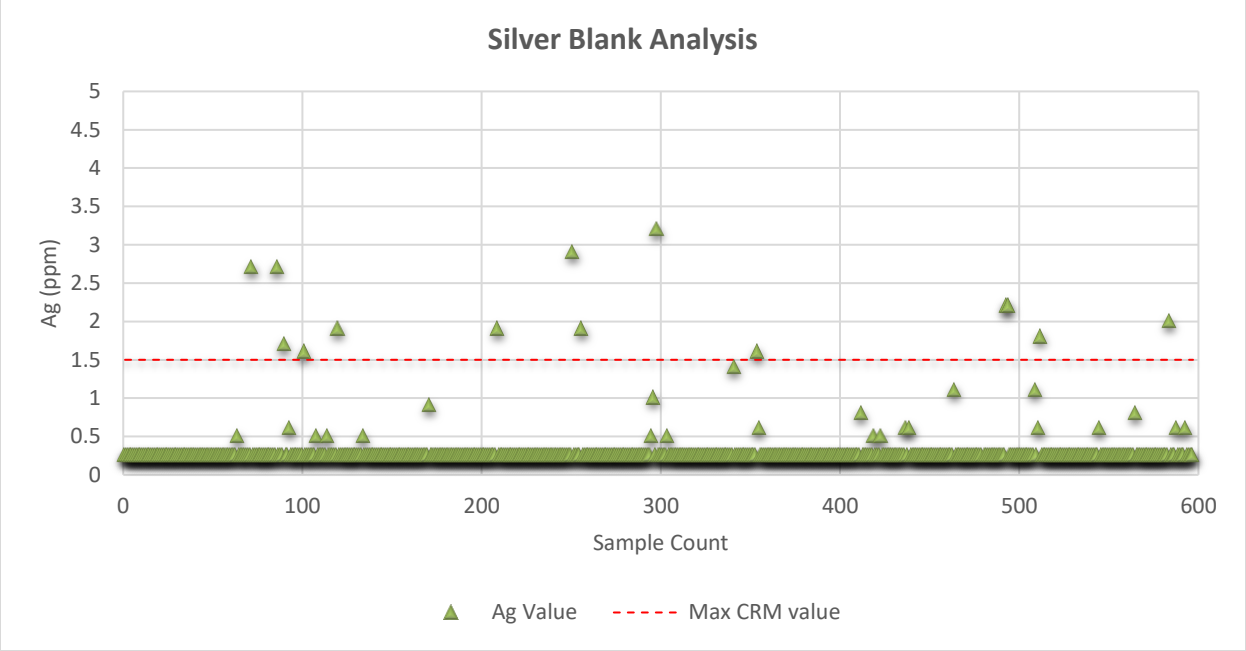


Figure 11-2: Silver Blank Analysis

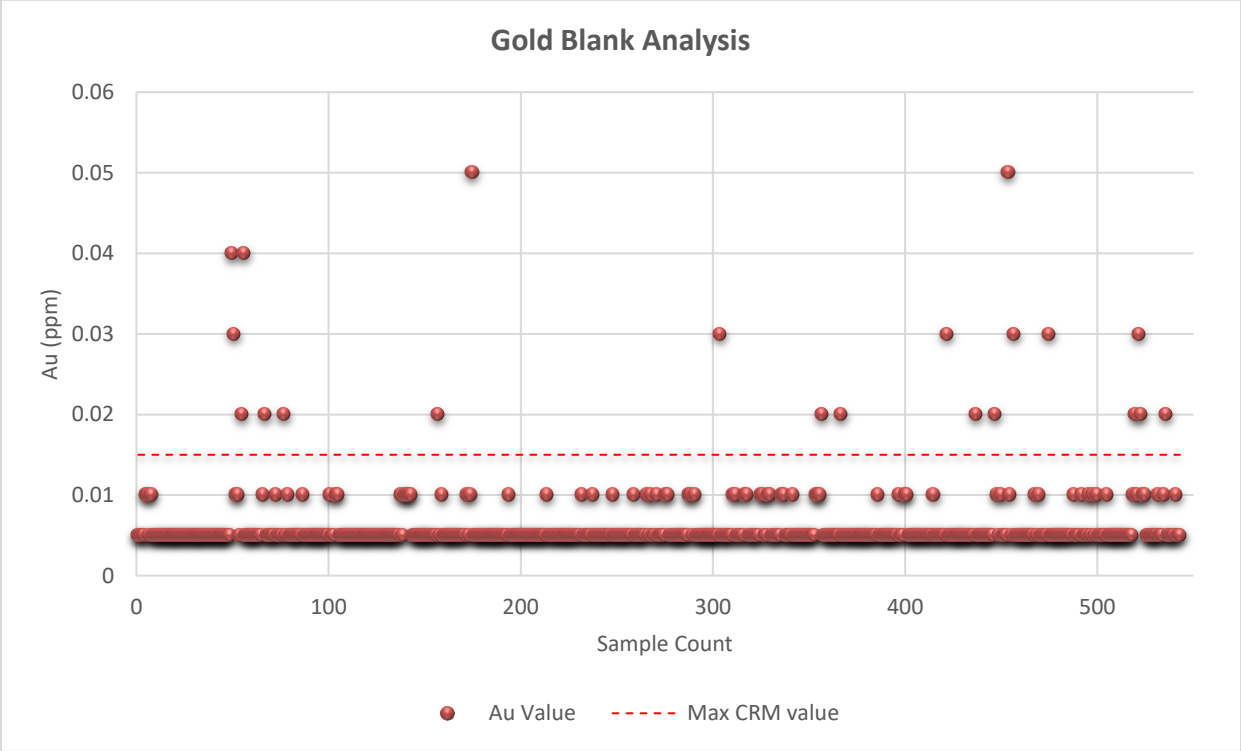


Figure 11-3: Gold Blank Analysis

11.2.2 Standard Analysis

Laboratory standards, like the CRMs employed by Konnex, play a crucial role in monitoring analytical consistency and detecting sample mix-ups. These CRMs possess a documented performance range (typically ±2 standard deviations from the mean), indicating expected behavior 95% of the time. This range is established through a rigorous process involving multiple laboratories analyzing the CRM in numerous assays (often exceeding 28). Importantly, standards are specific to analytical techniques. Within-set analysis assesses the standard's homogeneity and the lab's ability to consistently reproduce the method, while between-set analysis considers these factors alongside potential bias between labs and within sample subsets. Specific CRMs Used:

- 2017: MEG-AG-2 and MEG-AG-3 (skarn deposit, Nevada; MEG Labs, Reno) - Tolerances for gold, silver, copper, and zinc are summarized in Table 11-2 and Table 11-3 respectively.
- 2018: MEG-CU-1 - Copper, zinc, silver, and lead tolerances are summarized in Table 11-4.
- 2022: A106010X and A106013X - Metal tolerances are presented in Table 11-5 and Table 11-6.

Table 11-2: MEG-AG-2 Standard Tolerance and Average Grade Information

MEG-AG-2	Lab Avg.	Std. Dev.	-2σ	+2σ
Cu%	0.25	0.01	0.23	0.27
Zn%	11.24	0.29	10.66	11.82
Ag g/T	292.7	20.3	252.1	333.3
Au g/T	1.05	0.04	0.97	1.13

Table 11-3: MEG-AG-3 Standard Tolerance and Average Grade Information

MEG-AG-3	Lab Avg.	Std. Dev.	-2σ	+2σ
Cu%	0.23	0.01	0.21	0.25
Zn%	10.4	0.28	9.84	10.936
Ag g/T	2684	168	2348	3020
Au g/T	1.57	0.1	1.37	1.77

Table 11-4: MEG-CU-1 Standard Tolerance and Average Grade Information

MEG-CU-1	Lab Avg.	Std. Dev.	-2σ	+2σ
Cu%	0.48	0.02	0.44	0.52
Zn%	2.53	0.11	2.30	2.76
Ag ppm	25	1.33	2.34	29.66
Pb ppm	1016	49	918	1095

Table 11-5: A106010X Standard Tolerance and Average Grade Information

Standard	Lab Avg.	Std. Dev.	-2σ	+2σ
Cu%	0.215	0.11	0.193	0.237

Table 11-6: A106013X Standard Tolerance and Average Grade Information

Standard	Lab Avg.	Std. Dev.	-2σ	+2σ
Cu%	0.574	0.019	0.536	0.612
Ag ppm	4.07	0.415	3.24	4.90
W ppm	61	5	51	71
Mo ppm	310	34	242	377

Konnex employed Z-score plots to analyze their standards and ensure reliable assay data throughout the database. Each standard was plotted against its expected mean value, with ±2 and ±3 standard deviation lines demarcating acceptable ranges (Figure 11-4).

Konnex failure criteria occurs when a standard assay falls outside the ±3 standard deviation range. If the five samples preceding and following the failed standard exhibit anomalous concentrations of copper (Cu), gold (Au), or silver (Ag), a re-assay is triggered for all ten surrounding samples. A new standard is included with each re-assay batch.

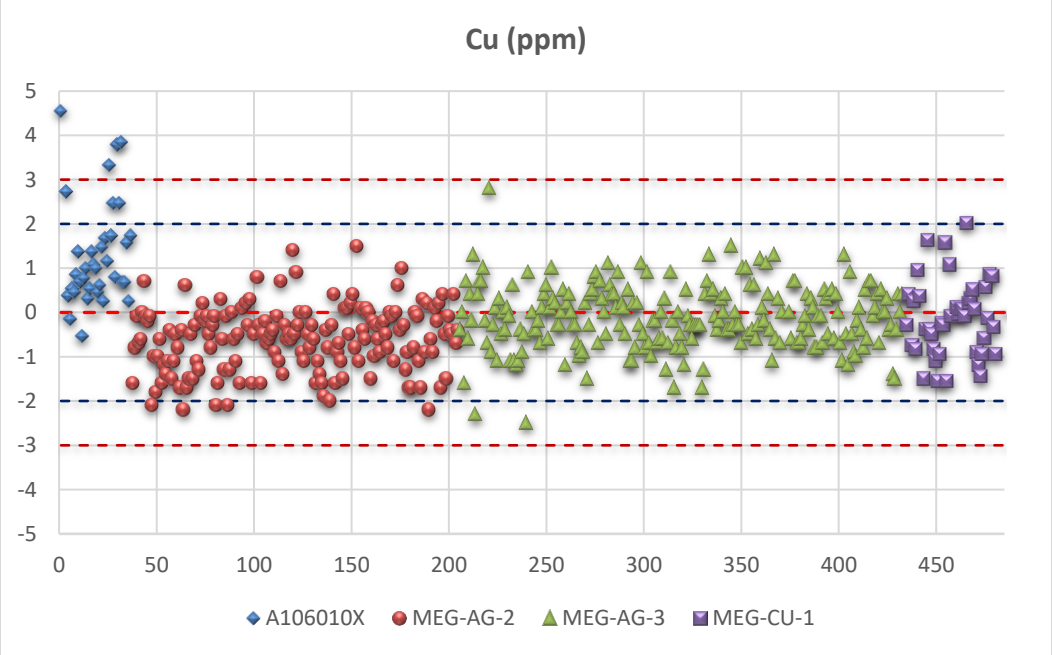


Figure 11-4: Z-Score Plot of Copper Standards

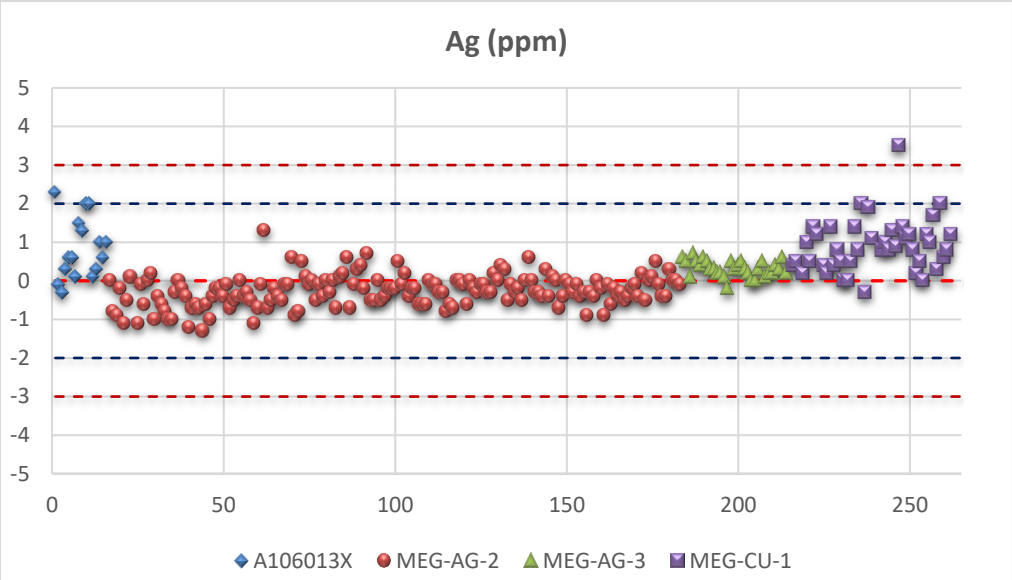


Figure 11-5: Z-Score Plot of Silver Standards

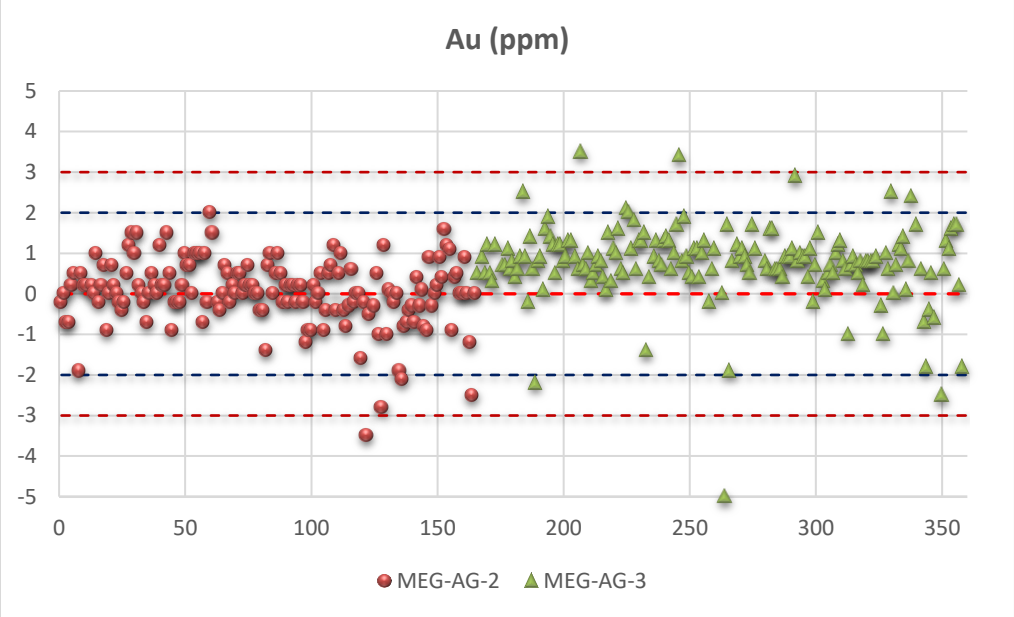


Figure 11-6: Z-Score Plot of Gold Standards

HRC's review of standards revealed a high degree of conformance to Konnex's established failure criteria. Only a minimal number of standards fell outside the acceptable ±3 standard deviation range. As per protocol, any samples associated with failing standards should be re-analyzed before incorporation into the drill hole database.

11.2.3 Duplicate Analysis

Duplicates are used to monitor sample batches for sample mix-ups, data variability due to laboratory error and sample homogeneity at each step of preparation. Sample duplicates should be inserted at every sample split during sample preparation, and they should not be placed in sequential order. To assess data quality for coarse and field duplicates, absolute relative difference (ARD) is calculated and plotted. This metric compares the magnitude of the difference between the original and duplicate sample values relative to their average. The acceptance criteria vary based on sample type:

- Coarse duplicates: 90% of ARD values should be less than or equal to 20%.
- Field duplicates: 90% of ARD values should be less than or equal to 25%.

Figures Figure 11-7 through Figure 11-9 illustrate the ARD analysis for field duplicates. The data reveals variability between elements, with silver demonstrating the highest concordance (85% of pairs ≤25% ARD), followed by copper (76% of pairs ≤25% ARD). However, gold data quality warrants further scrutiny, as only 60% of duplicate pairs meet the ≤25% ARD acceptance criterion. This suggests potential inconsistencies during field duplicate collection for both core and RC samples.

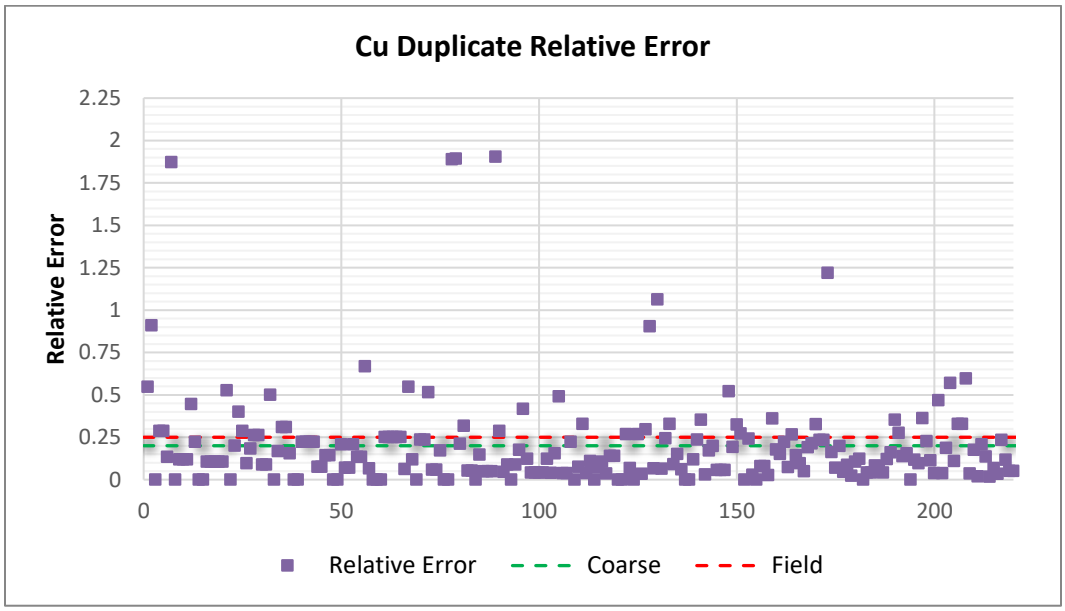


Figure 11-7: Absolute Relative Differences Plot for Copper Duplicates

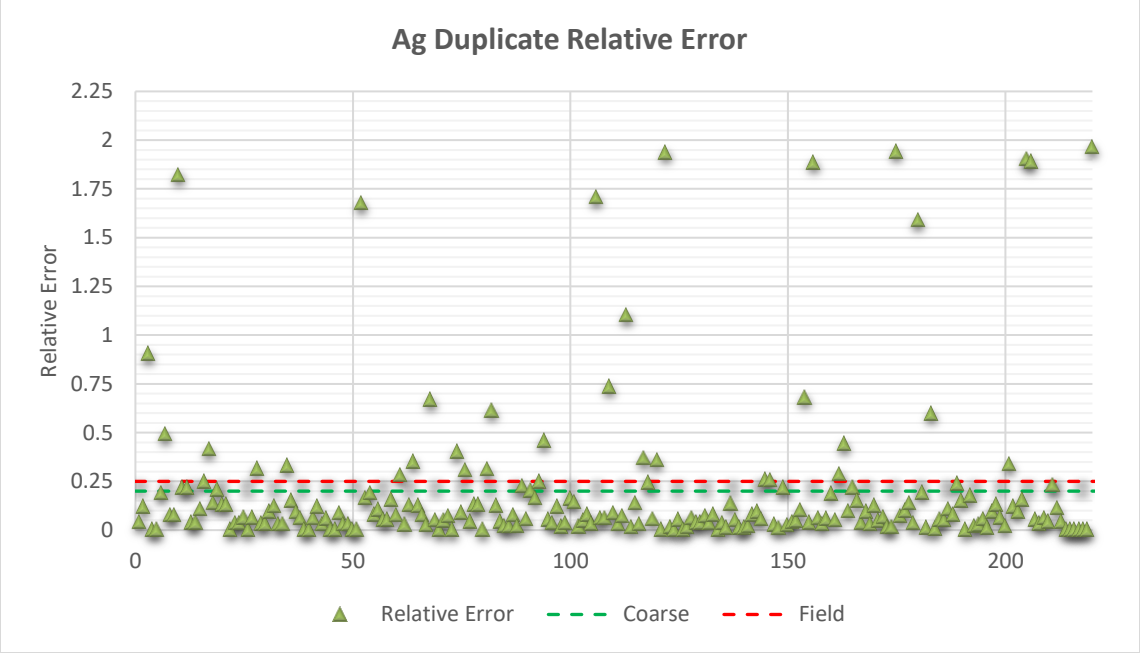


Figure 11-8: Absolute Relative Differences Plot for Silver Duplicates

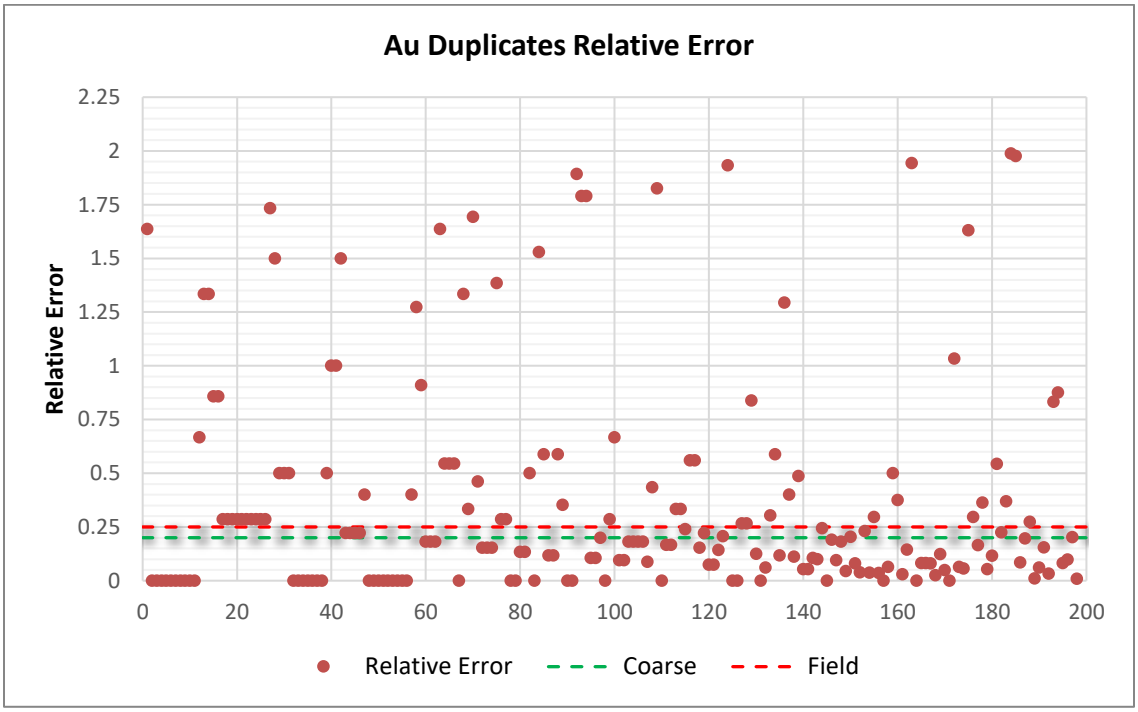


Figure 11-9: Absolute Relative Differences Plot for Gold Duplicates

11.3 Opinion on Adequacy

HRC's evaluation indicates that Konnex's sample preparation, security, and analytical procedures employed throughout the project generally adhere to acceptable industry standards. Consequently, the generated analytical data is considered adequate for inclusion in mineral resource estimation. The chosen sampling methods and densities are deemed appropriate, and the collected samples demonstrably possess sufficient quality to constitute a representative and unbiased data set.

The overall QA/QC program, including analysis of standards, blanks, and duplicates, identified minor areas for improvement. While the data quality is sufficient for resource estimation, implementing these improvements would further enhance data confidence. Specific focus should be directed towards strengthening the QA/QC program to ensure it consistently meets industry best practices across all aspects, including standard analysis, blank analysis, and field duplicate collection procedures for both core and RC samples. As part of this effort, Konnex should perform acceptance/rejection tests consistently using the same procedures throughout the program. Transparency and good relations with the laboratory are always necessary to ensure a successful QA/QC program.

12. DATA VERIFICATION

This section details the verification procedures employed by the qualified person to ensure the accuracy and reliability of the data used within this technical report. The verification process aims to confirm the data's origin, identify and address internal inconsistencies, and evaluate its alignment with external sources. Additionally, the effectiveness of the implemented Quality Assurance/Quality Control (QA/QC) program is reviewed. The following sections will outline the specific verification steps undertaken, discuss any limitations encountered during the process, and ultimately provide a professional opinion on the overall adequacy of the data for the purposes of this report.

12.1 Data Verification Procedures

A comprehensive verification process was implemented to ensure the accuracy and reliability of the data utilized within this technical report. This process encompassed several key steps:

12.1.1 Current Personal Inspection

Site Inspection and Data Oversight (March 2022 - June 2024): Zachary Black, SME-RM (qualified person - QP), served as the General Manager for Konnex between March 2022 and June 2024. In this role, Mr. Black directly oversaw critical aspects of data acquisition and management, including:

- Drilling Programs: Personal supervision of drilling programs to ensure adherence to established protocols and data collection procedures.
- Drill Hole Database Construction: Direct oversight of the construction and population of the drill hole database, minimizing the risk of errors during data entry.
- Data Management: Implementation and enforcement of data management protocols to ensure data integrity and consistency.
- Historical Drilling Validation: Active participation in the validation of historical drilling data, leveraging his experience and knowledge of the project history.

These combined efforts provided a robust verification framework for the exploration data used in this report.

Additional Activities (May 29, 2019):

A focused site visit was conducted on May 29, 2019, by J.J. Brown, P.G. (qualified person - QP). Discussions were held with Konnex personnel to gain a thorough understanding of the project, data collection procedures, and exploration history. Additionally, a personal visit to the project site and field office was conducted to evaluate field practices and data management protocols firsthand. This visit included:

- General Geologic Reconnaissance: Inspection of facilities, outcrops, and drill collar locations to verify consistency with reported data and assess geologic continuity.
- Core Review: Examination of core intervals across various lithologies and grade ranges for visual inspection and check sampling. This confirmed alignment between observed lithologies and logged information for most samples.
- Review of Procedures and Data Management: Discussions with Konnex staff regarding geologic models, data management protocols, drilling/sampling procedures, and QA/QC methods.

Field Observations: Field observations generally confirmed previous reports on project geology. Bedrock lithologies, alteration, and structures aligned with existing descriptions. No significant discrepancies were identified that could contradict the current geological interpretation.

Core Review: Core intervals were selected across various lithologies and grade ranges for visual inspection and check sampling. In most cases, core observations aligned with logged lithologies and assay values. However, some discrepancies were identified between logged lithologies and observed lithologies due to a 2018 re-logging effort focused on consistency with historical data. Konnex acknowledges the need to refine these re-logged lithologies within the database for improved accuracy. While this "lumping" of lithologies might have a minor impact on copper grade distribution, it is not considered significant for the current resource estimation.

12.1.2 Review of Historical Data and Internal Reports

Historical reports and internal company documents relevant to the project were reviewed in detail. This cross-referencing process helped identify potentially valuable supplementary information and verify the accuracy of existing geologic and analytical data.

12.1.2.1 Pre-Atlantic Pacific Exploration

Historical drilling data from various programs conducted before the development of the Atlantic Pacific Pit was evaluated for inclusion in the resource estimate and interpretations of potential mineralization zones. Here's a summary of the validation process:

U.S.B.M. Data (1943) was excluded from the resource estimate due to concerns about the accuracy of its historical mine grid for coordinate conversion. However, the data was used for interpretations of potential mineralization zones in areas that are difficult to target with surface drilling methods.

Data from the other companies (1962-1975) programs were considered adequate for resource estimation, despite some limitations:

- Limited Collar Verification: While direct verification of drill hole collars through physical inspection was not possible due to mining activity, a two-pronged approach was employed for indirect verification:
    - Historical Data Comparison: Collar coordinates were compared against historical company maps to assess their internal consistency.
    - Field Evidence Assessment: Despite the removal of much field evidence during mining, efforts were made to visit the collar locations and search for remnants of drilling activity, such as casing fragments in the ground. The presence of such remnants provided some confidence in the accuracy of the collar coordinates.
- Consistency with Konnex Data: The available data (assay certificates and drill logs) provided sufficient details regarding drill orientation and sample types. Additionally, the assay results from these programs generally agreed with nearby Konnex samples, which followed established analytical protocols.

12.1.2.2 Cambior Drilling Data (1995-1997)

The suitability of Cambior's drilling data (1995-1997) for resource estimation or further exploration was assessed by evaluating sampling procedures, security measures, and quality assurance/quality control (QA/QC) protocols. Information was compiled from two sources:

- Phil Van Angeren's "Geological Assessment and Exploration Proposal (2004)" for the Empire Mine Project.
- SRK Report (2017).

12.1.2.2.1 Sampling Procedures

Available information suggests that core samples were split and collected at 5-foot intervals, with adjustments made based on observed geological features. However, details regarding specific sampling methods employed, the qualifications of personnel involved, and the security measures taken during sample storage are limited.

12.1.2.2.2 Quality Assurance/Quality Control (QA/QC)

SRK assessed the implemented QA/QC protocols for the Cambior drilling data (1995-1997). While the use of Certified Reference Materials (CRMs) within the sample stream was identified, the specific types of standards employed remain unknown. Cambior reportedly developed five internal CRMs with varying target grades; however, their performance evaluation focused solely on gold, not copper, the primary element of interest in this project. Additionally, pulp duplicate samples were collected and re-assayed, demonstrating generally good correlation with the original assays, except for a limited number of samples from a single drill hole. To validate these findings, HRC replicated SRK's analysis of the QA/QC data.

12.1.2.2.3 Limitations and Considerations

Despite limitations in the available information on sampling methods, personnel qualifications, and sample security, as well as the unknown standard types used in CRMs and their focus on gold rather than copper, SRK's assessment of the Cambior drilling data's QA/QC protocols and HRC's subsequent verification provide confidence in the data's overall integrity. Consequently, the data is considered adequate for inclusion in mineral resource estimation.

12.1.2.3 Trio Drilling Data (2004)

The Trio drilling data (2004) details include sampling procedures and the implemented quality assurance/quality control (QA/QC) protocols.

12.1.2.3.1 Sampling Procedures

RC cuttings were collected at regular intervals of 1.5 meters (5 feet) during the drilling program. Two samples were collected from each interval, with individual sample weights ranging between 900 grams and 5,440 grams depending on the amount of material recovered. Unfortunately, there is no documented information regarding the specific method used to split these collected samples.

12.1.2.3.2 Quality Assurance/Quality Control (QA/QC)

An evaluation of the QA/QC protocols revealed that a subset of samples was re-assayed at a different laboratory using an alternative analytical technique. However, no details were found concerning the use of blanks or standard reference materials within the sampling process.

The analytical methods employed differed slightly between the two laboratories involved:

American Assay Laboratories (AAL) utilized a multi-acid digestion followed by a 69-element ICP analysis for all samples. This analysis provided total copper values. Additionally, they conducted a separate acid-soluble copper and zinc assay using a sulfuric acid digestion technique. Gold analysis was performed independently using fire assay with an atomic absorption finish. Samples exceeding 10,000 ppm copper and zinc underwent further analysis by atomic absorption.

Loring Laboratories analyzed all their samples using a multi-acid digestion followed by a 32-element ICP analysis. Similar to AAL, gold was assayed separately by fire assay with an atomic absorption finish. Loring Laboratories also analyzed samples exceeding 10,000 ppm copper and zinc via atomic absorption.

The key findings from the QA/QC program indicated good correlation between the re-assay results from both laboratories for most elements analyzed. However, Loring Laboratories consistently reported higher assay values for low-level gold samples (less than 0.1-0.3 g/t Au). Additionally, a small number of samples exhibited discrepancies, particularly for some high-grade silver results and low-grade gold results. Sample mislabeling during collection or processing is a potential explanation for these observed inconsistencies.

12.1.2.3.3 Limitations and Considerations

The absence of information regarding the sample splitting method and the lack of blank or standard reference material utilization introduce some limitations to the data's overall robustness. However, the good correlation between re-assay results from two independent laboratories using different analytical techniques provides confidence in the overall data integrity. Considering these limitations and the positive aspects of the QA/QC protocols, the Trio drilling data is considered adequate for inclusion in mineral resource estimation.

12.1.2.4 Journey (2006)

ALS Chemex Laboratories of Reno, Nevada performed the primary analysis on all the Journey drill program samples using a two-acid digestion, 41-element ICP analysis. For samples with copper and zinc concentrations exceeding 10,000 ppm, a separate acid-soluble assay using sulfuric acid digestion was conducted.

Instead of employing blanks or certified reference materials for quality control, all samples were further analyzed for total copper and zinc content using atomic absorption spectrometry as a form of duplicate/check assay.

Gold and silver analyses were performed independently using fire assay with an atomic absorption finish (van Angeren, 2014). A total of 2364 samples were analyzed for copper and zinc using both the two-acid digestion ICP finish and a fire assay AAS finish. The fire assay method has a higher lower detection limit (Cu, Zn) compared to the ICP method (100 ppm vs 4-16 ppm).

The copper results from fire assay correlated well with ICP, showing slightly higher concentrations overall. While zinc results correlated well above 0.3%, fire assay performed poorly for zinc concentrations below this threshold. Consequently, fire assay is not considered a suitable method for zinc analysis in this project.

12.1.3 Musgrove (2011)

12.1.3.1 Fire Assay Check Program

In 2011 Musgrove conducted an additional check assay program to further evaluate the Journey data. This program re-analyzed a subset of 162 samples for specific elements:

- Copper (Cu), Zinc (Zn), and Silver (Ag) were re-assayed using a 4-acid digestion ICP method.

- Gold (Au) was re-determined using Fire Assay with Atomic Absorption Finish.

A comparison with the original assays (2006) revealed the following:

- Copper (Cu) and Zinc (Zn): Good correlation overall, with some outliers potentially due to sample mix-ups during preparation.
- Silver (Ag): Broadly correlates with the 2006 data, but the check assay program yielded higher average silver concentrations.
- Gold (Au): Limited data above 0.3 g/t Au in the re-analyzed samples hinders a conclusive evaluation of the correlation with the original assays.

12.1.3.1.1 Drilling Sample Analysis

ALS Laboratories performed the primary analysis on all the Musgrove drill program samples using a four-acid digestion followed by a 33-element ICP analysis for all samples. Additionally, ALS performed acid-soluble assays (AS) for copper and zinc using sulfuric acid digestion. Gold analysis was conducted via fire assay with an atomic absorption finish.

Musgrove employed several Quality Assurance/Quality Control (QA/QC) procedures to ensure data accuracy and precision. Certified Reference Materials (CRMs) were used to monitor the consistency of analytical results with known standards. A total of 87 CRMs were included in the sample stream, consisting of two specific standards:

- 52 (MEG-Ag-2)
- 35 (MEG-Ag-3)

Blanks were strategically inserted throughout the sample stream to assess potential contamination during sample preparation and analysis. Musgrove utilized a total of 47 blanks, sourced from rhyolite, and inserted them every 50 feet within the sample stream.

Duplicates were analyzed to evaluate the precision of the analytical methods by comparing results to original. A total of 145 samples, believed to be from reject RC material (van Angeren, pers. comm., 2017), were duplicated.

Finally, samples exceeding specific thresholds for copper, zinc, lead, and silver were re-analyzed using ore-grade atomic absorption for further verification (van Angeren, 2014).

The QA/QC program yielded mixed results. Analysis of blanks indicated minimal contamination, with the highest zinc concentration reaching only 104 ppm. Duplicate analysis suggested good precision, although the number of samples with significant gold content (above 0.1 g/t) was limited (14). MEG-Ag-2 CRM analysis of the 52 samples revealed acceptable accuracy for copper and zinc, with copper showing a slight underreporting (around 4%). However, two gold assays (EM11-06 and EM11-07) fell outside acceptable lower limits. Additionally, a potentially mislabeled sample (EM11-05) was identified with a high silver value (452 ppm). Silver grades also exhibited a minor underreporting trend (approximately 6%). While a separate analysis of the 35 MEG-Ag-3 CRMs showed acceptable accuracy for copper, silver, and zinc. However, five gold assays displayed significant underreporting with no explanation provided.

12.1.4 HRC Exploration Database Validation

HRC conducted a rigorous verification process on the exploration drillhole data received from Konnex to ensure its accuracy and consistency for this report. This process involved three key steps:

- Mechanical Audit: HRC checked the data for missing entries, inconsistencies in data format, and potential data entry errors.
- Geological Information Validation: HRC compared the geological information in the database with corresponding paper logs to identify any discrepancies.
- Assay Value Validation: HRC compared the assay data in the database with the original assay certificates provided by Konnex to verify the accuracy of reported values for all elements of interest.

12.1.4.1 Data Source Verification

The database was reconstructed from original assay certificates by Konnex. To confirm accuracy, HRC conducted a random manual check of 10% of the data. This validation indicated an error rate of less than 1%.

12.1.4.2 Survey Data

Collar coordinates were compared with surface topography by HRC and deemed adequate. While down-hole surveys were missing for many drillholes, their locations were considered acceptable based on comparisons with surrounding surveyed holes.

12.2 Opinion on Adequacy

Based on the verification procedures conducted by HRC, the exploration data used in this report is considered adequate for mineral resource estimation.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

The copper-gold-zinc-silver mineralization at the Empire Project falls into the skarn hosted, polymetallic category. The oxides (e.g. carbonates such as malachite and azurite) and sulfide (chalcopyrite and chalcocite) mineralization is developed to varying degrees within exoskarn in limestone and endoskarn in porphyry.

Metallurgical studies have been performed on samples from the Empire Mine by several companies since 1997. Although the location of the majority of the past metallurgical sampling is not well defined in the available reports, it appears that the majority of the work has been undertaken on material that is within 100 feet of surface.

Konnex acquired the Empire Project in 2017 and initiated systematic development of the project. Metallurgical testing has resulted in the development of a techno-economic process flowsheet. The historical and recent test work are discussed in this section.

13.1 Historical Studies

The emphasis of metallurgical studies from 1997 to 2022 was directed at scoping level testing of several processing options including the following:

- Two-stage agitated leach process (acid leach/cyanide leach) for the recovery of copper and precious metals.
- Two-stage acid heap leach followed by cyanide agitated leach process for recovering copper and precious metals.
- One-stage agitated leach process using ammonium thiosulfate (ATS) to recover copper, precious metals, and zinc.

13.1.1 Cambior (1997)[1-4]

Cambior USA tested a two-stage process (acid leach/cyanide leach) for the recovery of copper and precious metals in 1997. The study concluded that the optimum process was agitated acid leach for copper recovery, washing of the leach residue and cyanidation process for precious metal recoveries. The primary grind was determined to be between P_{80} of 50 to 100 mesh. The projected recoveries for Cu, Au, and Ag were +/- 80%, +/- 80%, and +/- 50% to 60% respectively. The flowsheet had SX/EW for copper and Merrill Crowe for gold and silver recoveries.

13.1.2 KCA (2005)[5]

KCA performed bottle roll and column tests on a composite sample assaying 0.8% Cu, 0.003 opt Au, and 0.30 opt Ag for Trio Gold Corp. Column acid leach tests were performed on “as received” and P_{100} of 1.5-inch particle size. Copper extraction was 31% on “as is” material in 82 days and 61% on P_{100} of 1.5-inch crush size in 89 days. Ferric sulfate was added during cure and no significant slump was noted in the tests. Assay-by-size on leach residue indicated 0.5-inch crush size was optimum for copper extraction. The bottle roll cyanidation tests at P_{80} of 200 mesh indicated 42% of gold and 92% of silver extraction was obtained in the test.

13.1.3 KCA (2013)[6]

KCA performed bottle roll tests on two composite samples (exoskarn and FeOx-clay breccia) for Boxxer Gold Corp. They tested both acid heap leach and agitated leach for copper on exoskarn sample assaying 0.062g/t Au, 21.05g/t Ag, and 1.582% Cu and agitated cyanidation for gold and silver on FeOx-clay breccia sample assaying 0.273g/t Au, 22.59g/t Ag, and 1.98% Cu. Acid bottle roll tests on exoskarn composite with 10g/L sulfuric acid extracted 52% copper in 240 hours at P_{80} of 19mm and 95% copper at P_{80} of 75 microns. The acid consumptions were 22.92kg/t and 34.98kg/t, respectively.

The cyanidation leach test was performed on FeOx-breccia sample at P_{80} of 75 micrometers for 96 hours with 1g/L NaCN. The gold and silver extractions were 76% and 37% respectively. The NaCN consumption was high (4.19kg/t) due to the presence of acid/cyanide soluble copper.

13.1.4 Mineral Technologies Inc (2018)[7]

Mineral Technologies Inc performed bottle roll acid leach tests, open-cycle column leach tests, and bottle roll cyanide leach tests on four composite samples from the mine site. The four composites were: (1) Exoskarn, (2) Granite Porphyry/Endoskarn, (3) Magnetite Skarn, and (4) FeOxide-Clay Breccia.

The test results indicated the following:

- The head analyses of the composite samples were 0.416% to 0.585% Cu, 0.211% to 0.664% Zn, 0.082 to 0.461g/t Au, and 10.4 to 26.7g/t Ag.
- The acid soluble copper in the samples ranged from 39.8% for FeOxide-clay breccia to 90.5% for exoskarn and endoskarn.
- The copper extraction in the bottle roll tests at P100 of 200 mesh and 24 hours of leach time were 68% to 81% for samples 1 to 3 and only 32% for FeOxide sample.
- The zinc extractions range from a low of 12% for FeOxide sample to a high of 63% for exoskarn sample.
- The acid consumption was extremely high for exoskarn (64.1kg/t) and reasonable (23kg/t to 32kg/t) for the other composites.
- Eight open-circuit column tests were performed at a P100 of 1 inch for 118 to 163 days. The copper and zinc extractions ranged from 41.4% to 81.4% and 13.4% to 62.9%, respectively. The acid consumption was high (50kg/t to 60kg/t) except for FeOxide-clay breccia sample which also exhibited the lowest metal extractions.
- Bottle roll tests were performed on column leach residue for magnetite skarn and FeOxide-clay breccia. Excellent gold extraction (+/- 90%) was obtained in the tests. However, the silver extraction was poor (29% to 59%).

These results indicated that heap at 1 inch crush size would recover +/- 75% Cu and +/- 50% Zn with acid consumption of +/- 50kg/t. There was no advantage to crushing finer than 1 inch or agglomerating the material.

13.1.5 AuRIC (2020)[8]

The laboratory performed tests for Konnex Resources to compare extraction of gold from sample of magnetite ore using two lixiviants, namely sodium cyanide and ammonium thiosulfate (ATS). The head analyses of the sample was 12.78g/t Au and 13.43g/t Ag.

Both ATS and NaCN had fast kinetics and extracted 95% to 97% of gold and 65% to 70% of silver. The ATS consumption was estimated to be 4%.

13.1.6 Forte Analytical (2022)[9]

Forte Analytical (Forte) performed two-stage leach tests (acid-ATS) on ROM and core reject samples of endoskarn, exoskarn, and magnetite ore types from the Empire Mine. The highlights of the test work indicated the following:

- The samples assayed 0.17% to 0.39% Cu, 0.02 to 0.11 opt Au, and 0.10 to 0.66 opt Ag.
- The sequential analyses of copper indicated 87.5% to 92.2% acid soluble copper for skarn samples and 53.8% for magnetite sample. The remaining copper is present as primary copper.
- The mineralogical investigation indicated chrysocolla as the primary copper in all ore types. Traces of bornite and cuprite were also present in the samples.
- Hedenbergite and epidote are the major minerals in the samples. Both minerals are high acid consumers.
- Crushability work index and abrasion index determinations indicated that the ores are soft and non-abrasive.
- Two-stage bottle roll tests indicated that 35% to 80% of copper and 5% to 72% of zinc was extracted in the acid leach step. The ATS leach on acid leach residue material extracted 92% to 96% of gold, 57% to 94% of silver, 37% to 52% of copper, and 12% to 38% of zinc.
- Four column tests each were performed on endoskarn and exoskarn samples crushed to P100 of 1 inch for 105 days. Copper extractions ranged from 78.6% to 89.8% for endoskarn and 65.3% to 89.5% for exoskarn samples. The corresponding zinc extractions were 52.4% to 70.9% and 49.2% to 87.5%, respectively. The acid consumption was relatively high and ranged from 96 lbs/st to 262 lbs/st.
- ATS leached both gold and silver with good extractions.

13.1.7 CSA Global (2022)[10]

The investigators reviewed the drilling data and metallurgical results and developed a lithiogeochemical classification and rock-type definition for the deposit. The investigation resulted in 11 lithiochemical units.

13.1.8 Ausenco (2022)[11]

Ausenco reviewed the ammonium thiosulfate (ATS) test work from two laboratories, AuRIC and Forte Dynamics. They concluded that the gold can be leached well from all lithiological samples with recovery of over 80%. Copper can be leached to some degree with thiosulfate, but higher copper recoveries can be achieved by standard acid leaching methods.

Ausenco raised several issues regarding the use of ATS:

- ATS leach is a much more complex chemical system and will need to be well understood to gain confidence in its operability at an industrial level.
- The ability to recover all of the metals from the leach solution will be the key to generating an economic flowsheet.
- Recovery of gold from ATS solution has not been evaluated. Neither is the potential of recycling of ATS after metals are recovered.

13.1.9 Conclusions

The historical metallurgical studies indicated the following:

- Copper, and to some extent zinc, minerals are leachable with sulfuric acid. The finer the particle size, the higher the copper extraction.
- The acid consumption will be high due to the presence of calcium-bearing minerals.
- Gold and silver can be recovered by leaching with sodium cyanide or ammonium thiosulfate (ATS). Some copper can also be extracted by ATS.

13.2 Forte Analytical Recent Studies – Phase I (2022-2023)[12]

Konnex Resources Inc. contracted Forte Analytical (Forte) to undertake metallurgical test work with the primary objective of developing a techno-economically viable process flowsheet for recovering copper and precious metals from Empire Mine ore without using sodium cyanide.

Forte received ½ core from mine holes drilled at site in 2022. The location and description of these holes are given in the metallurgical report.

There are three geological domains – Endoskarn, Exoskarn, and Magnetite. Since Endoskarn and Exoskarn have similar metallurgical responses in historical test work, a decision was made to prepare two master composites, Endo/Exo and Magnetite. A high-grade Magnetite composite, assaying over 2 g/t Au (0.0584 opt) was also received for the study. In addition, +2 inch rocks were received for crusher work index determination.

Each interval was crushed to nominal 3/4 inch and split into two fractions. One fraction was saved, and the other fraction was used for preparing a composite sample. This procedure was utilized for both Endo/Exo and Magnetite composites.

The composite samples were blended, and a split taken out for abrasion index determination. The remaining composite samples were crushed to P100 of 6 mesh, blended, and split into 1-kg and 10-kg charges for metallurgical studies.

One 10-kg sample of each composite was taken for Bond's ball mill work index (BWi) determination. One 1-kg charge was pulverized, and splits were submitted for head analyses and whole rock analyses.

13.2.1 Head Analyses

The analytical results for the master composites are summarized in Table 13-1 and Table 13-2.

Table 13-1: Head Analyses of Composite Samples

Element	Endo/Exo	Magnetite
Au, ppm		
Assay #1	0.16	0.14
Assay #2	0.12	0.15
Average	0.14	0.145
Ag, ppm		
Assay #1	22.7	13.7
Assay #2	14.7	15.0
Average	18.7	14.3
Cu_{Total}	0.43	0.75

Table 13-2: Whole Rock Analyses of Endo/Exo and Magnetite Master Composites

Element	Endo/Exo	Magnetite
SiO_2	53.31	43.95
Al_2O_3	10.04	5.92
Fe_2O_3	12.13	34.27
CaO	14.29	6.50
MgO	1.62	0.59
Na_2O	0.91	0.07
K_2O	2.62	0.59
TiO_2	0.37	0.29
MnO	0.38	0.54
SrO	0.02	<0.01
BaO	0.07	0.03
Cr_3O_3	0.01	0.02
P_2O_5	0.30	0.55
ppm		
Ag	19.4	19.4
As	112	316
Ba	606	286
Bi	<10	90
Cd	23	51
Co	23	41
Cr	96	98
Cu	4300	7400
Mo	62	177
Ni	25	45
Pb	138	177
Sr	261	87
V	62	77
W	174	193
Zn	3200	4600

These results indicated the following:

- The Endo/Exo master composite assayed 0.43% Cu, 0.14 ppm Au and 18.7 ppm Ag.
- The Magnetite master composite assayed 0.75% Cu, 0.145 ppm Au and 14.3 ppm Ag.

13.2.2 Comminution Tests

The comminution tests were performed on the two master composites. These tests included crusher work index, abrasion index, and Bond's ball mill work index. In addition, indirect ball mill work indices were evaluated for seven variability samples.

The comminution test results are summarized in Table 13-3 and Table 13-4.

Table 13-3: **Comminution Test Results**

Parameter	Endo/Exo Composite	Magnetite Composite
Crusher Work Index (CWi, kwh/st)	14.23	15.63
Abrasion Index (Ai)	0.0773	0.1566
Bond Ball Mill Work Index (BWi, kwh/st)	15.29	14.08

Table 13-4: Indirect Bond's Ball Mill Work Indices

Sample	Ore Type	BWi (kwh/st)
Interval 2200102 Hole KXD 22-03	Exo	14.33
Interval 2200101/103 Hole KXD 22-03	Exo	15.16
Interval 2200298/667 Holes KXD 22-06/09	Endo	13.71
Interval 2200104/596 Holes KXD 22-03/08	Endo	14.55
Interval 2200291 Hole KXD 22-06	Endo	11.84
Interval 2200679 Holes KXD 22-01/09	Endo	10.66
Interval 2200127/128 Hole KXD 22-03	Endo	18.39

These results indicated the following:

- Crusher work index (CWi) was 14.23 kwh/st for Endo/Exo composite and 15.63 kwh/st for Magnetite composite.
- Abrasion index (Ai) was 0.0773 for Endo/Exo composite and 0.1566 for Magnetite composite. Ai of more than 0.3 is considered to be abrasive.
- Bond's ball mill work index (BWi) was 15.29 kwh/st for Endo/Exo composite and 14.08 kwh/st for Magnetite composite.

The indirect BWi for the variability samples ranged from 11.84 to 18.39 kwh/st. The majority of the samples had a BWi of less than 15.29 kwh/st.

13.2.3 Gravity Testing

A series of gravity tests were performed with the primary objective of recovering gold. The gravity tails were floated. These tests were performed with the Endo/Exo and Magnetite Master Composites (MC) at varying grind sizes.

The test flowsheet is shown in Figure 13-1. The test results are summarized in Table 13-5.

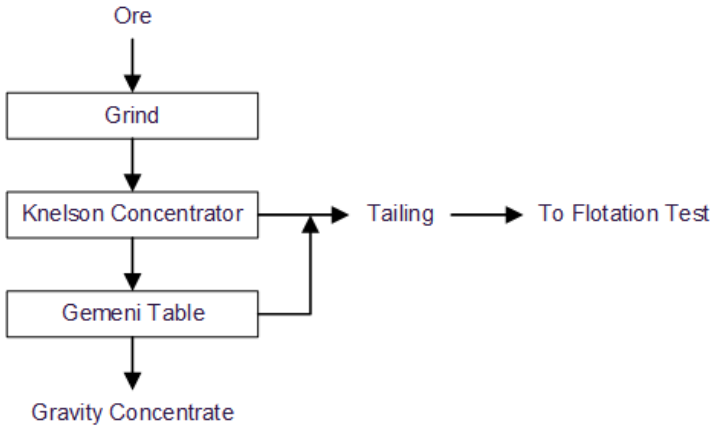


Figure 13-1: **Flowsheet for Gravity/Flotation Testing of Master Composites (Tests 1-8)**

Table 13-5: Gravity Test Results for Endo/Exo and Magnetite Master Composites

Test No.	Composite	Grind Size P_{80} Mesh	Cleaner Gravity Recovery %				Gravity Concentrate Grade			Calculated Feed Grade		
			Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
1	Endo/Exo MC	65	1.3	16.7	9.1	5.9	2.05	131	2.11	0.16	19	0.47
2	Endo/Exo MC	100	0.6	10.8	3.9	3.8	2.75	156	2.83	0.16	24.5	0.46
3	Endo/Exo MC	150	0.2	15.0	4.1	1.9	14.4	534	5.58	0.15	20.7	0.47
4	Endo/Exo MC	200	0.1	15.1	2.1	1.9	29.3	479	10.0	0.17	19.5	0.44
5	Magnetite MC	65	1.3	5.8	1.3	0.4	0.71	17.3	0.21	0.16	16.9	0.65
6	Magnetite MC	100	1.6	6.4	5.4	0.5	0.61	40.6	0.20	0.15	11.9	0.67
7	Magnetite MC	150	0.7	4.5	3.6	0.2	0.87	73.7	0.20	0.13	14.1	0.67
8	Magnetite MC	200	0.2	0.7	0.2	0.1	0.64	14.2	0.24	0.14	13.6	0.68

These results indicated the following:

- The feed gold grade of the two composites was extremely low (± 0.15 g/t Au or 0.0044 opt Au).
- The cleaner gravity concentrate recovered ± 15% of the gold for Endo/Exo MC and 0.7% to 6.4% of the gold for Magnetite MC.
- The silver recoveries were also low (< 9%) for the two composites.
- The higher the gravity concentrate grade for copper, the higher the gold grade.

Additional tests were performed with the high-grade Magnetite composite assaying over 2 g/t Au (0.0584 opt).

Upon completion of the initial test work, additional test work was performed with high-grade Magnetite composite, as shown on the test flowsheet Figure 13-2. The results for this testing are summarized in Table 13-6.

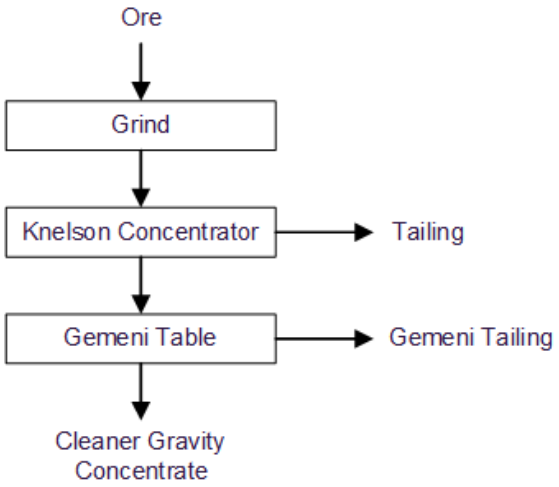


Figure 13-2: Gravity Testing Flowsheet for High Grade Magnetite Master Composites (Tests 9-11)

Table 13-6: Gravity Test Results for High Grade Magnetite Master Composite

Product	Assay			Distribution %			
	Au (g/mt)	Ag (g/mt)	% Cu	Wt	Au	Ag	Cu
Test 9: Grind P_{80} = 65 mesh							
Gemeni Conc	11.4	12.0	0.08	3.5	18.9	6.1	2.2
Gemeni Tail	4.48	<7.0	0.12	8.1	17.1	4.1	7.3
Knelson Conc	6.57	6.1	0.11	11.6	36.0	10.2	9.5
Knelson Tail	1.53	7.0	0.14	88.4	64.0	89.8	90.5
Cal Feed	2.11	6.9	0.13	100.0	100.0	100.0	100.0
Test 10: Grind P_{80} = 100 mesh							
Gemeni Conc	14.4	13.0	0.08	2.7	16.8	2.4	1.9
Gemeni Tail	3.19	15.0	0.14	8.5	11.7	8.6	10.8
Knelson Conc	5.89	14.5	0.13	11.3	28.5	10.9	12.8
Knelson Tail	1.87	15.0	0.11	88.7	71.5	89.1	87.2
Cal Feed	2.32	14.9	0.11	100.0	100.0	100.0	100.0
Test 11: Grind P_{80} = 150 mesh							
Gemeni Conc	12.5	36.0	0.09	2.4	15.9	8.1	1.6
Gemeni Tail	1.64	13.0	0.10	21.8	19.3	27.2	16.2
Knelson Conc	2.70	15.2	0.10	24.1	35.2	35.3	17.8
Knelson Tail	1.58	8.9	0.15	75.9	64.8	64.7	82.2
Cal Feed	1.85	10.4	0.13	100.0	100.0	100.0	100.0

These results indicated the following:

- The Gemeni concentrate assayed 11 to 14 g/t (0.32 to 0.408 opt) and recovered ± 16% of the gold. The low-grade of the concentrate indicates that the gold is not liberated.

The gravity test results indicated that the ore was not amenable to gravity concentration.

13.2.4 Ammonium Thiosulfate Leaching

Konnex wanted an environmentally friendly lixiviant for precious metal leaching. Hence, sodium cyanide could not be used. A laboratory in Salt Lake City, AURIC Met Lab, was successful in extracting gold using ammonium thiosulfate (ATS).

Forte, in cooperation with AURIC Met Lab, evaluated thiosulfate as an alternative to sodium cyanide for leaching of precious metals.

The test work was undertaken for both Endo/Exo and Magnetite composites. The ores were ground to varying grind sizes and leached for 8 hours in bottles at ± 40% solids. The slurry was adjusted for pH of 9.9 with ammonium hydroxide and varying amounts of ammonium thiosulfate was added prior to leaching. Kinetic samples were collected. The ATS leach recovered gold, silver, and some copper.

The test results are summarized in Table 13-7 to Table 13-10.

Table 13-7: Ammonium Thio-Sulfate Leach (ATS) at Varying Grind Sizes for Endo/Exo and Magnetite Master Composites

Test No.	Grind Size P_{80} Mesh	Extraction % (8 Hrs Leach)			Residue Grade			Calculated Head			ATS Consumption (kg/mt)
		Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu	
Endo/Exo MC											
1	65	93.7	90.9	43.1	0.03	1.4	0.235	0.47	15.0	0.413	55.6
2	100	93.9 (78.6)	84.5 (86.1)	43.0 (43.0)	0.03	2.6	0.239	0.49	16.8	0.419	42.1
3	150	94.7	65.3	42.0	0.03	7.5	0.237	0.57	21.7	0.409	48.4
4	200	92.2	59.2	42.1	0.04	8.3	0.238	0.51	20.2	0.412	46.9
Magnetite MC											
5	65	92.4	43.4	19.7	0.04	10.2	0.452	0.53	18.0	0.601	49.5
6	100	94.6 (79.3)	49.9 (46.2)	20.5 (20.5)	0.03	7.7	0.54	0.56	15.4	0.68	49.5
7	150	94.9	25.6	22.7	0.03	21.5	0.49	0.59	28.9	0.63	47.9
8	200	89.9	65.7	27.5	0.05	5.0	0.506	0.50	14.5	0.697	30.1

Note: Leach % Solids = 40. Numbers reported in parentheses are calculated recoveries based on feed acid residue assays.
ATS Concentration: 80 g/L

Table 13-8: Effect of ATS Concentration on Metal Extraction at P_{80} of 65 mesh for Endo/Exo and Magnetite Master Composites

Test No.	ATS Concentration (g/L)	Extraction % (8 hrs Leach)			Residue Grade			Calculated Head			ATS Consumption (kg/mt)
		Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu	
Endo/Exo MC											
9	40	85.9	69.4	25.7	0.03	4.2	0.2935	0.21	13.6	0.3948	38.3
10	60	94.4	85.7	37.9	0.02	2.2	0.2563	0.36	15.6	0.4129	24.9
11	80	92.4	41.1	38.1	0.03	19.7	0.2550	0.39	33.5	0.4118	34.1
Magnetite MC											
12	40	81.2	33.8	10.1	0.04	12.2	0.600	0.21	18.4	0.6300	37.0
13	60	86.0	44.2	16.9	0.06	8.7	0.549	0.43	15.5	0.6609	27.6
14	80	89.4	37.2	21.6	0.04	12.3	0.500	0.38	19.6	0.6400	25.5

Note: Leach % Solids = 40. Tests 11 and 14 were run at 50% solids.
Grind Size, P_{80} = 65 mesh

Table 13-9: Effect of Recycling ATS on Metal Extraction at P_{80} of 100 Mesh for Endo/Exo and Magnetite Composite

Cycle No.	Extraction %			Residue Grade			Calculated Head			ATS Consumption (kg/mt)
	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu	
Endo/Exo Composite										
1	92.7	76.3	43.0	0.04	4.9	0.2664	0.55	20.5	0.4670	43.6
2	92.1	74.6	25.4	0.03	5.0	0.2730	0.38	19.8	0.3663	42.3
3	91.1	82.3	41.2	0.03	3.7	0.2793	0.37	20.8	0.4747	48.0
4	92.1	64.3	41.3	0.03	7.3	0.2715	0.38	20.4	0.4623	31.7
5	94.6	64.4	16.4	0.02	5.9	0.2578	0.37	16.6	0.3086	26.1
6	93.0	70.4	13.0	0.02	3.6	0.2753	0.28	12.2	0.3165	20.2
Magnetite Composite										
1	89.7	41.4	20.4	0.06	13.1	0.5738	0.58	22.3	0.7221	56.4
2	89.4	25.8	20.3	0.03	27.5	0.5767	0.47	37.1	0.7237	33.0
3	80.9	48.4	18.7	0.06	9.5	0.6018	0.31	18.4	0.7407	49.9
4	91.4	41.3	20.3	0.04	10.5	0.5933	0.46	17.9	0.7448	32.6
5	70.2	31.4	4.0	0.10	10.7	0.5761	0.34	15.6	0.5999	31.7
6	91.3	45.2	7.4	0.04	6.6	0.6048	0.46	12.0	0.6531	17.8

Note: Leach Time: 8 Hours

Table 13-10: ATS Testing of Variability in Flotation Tailing Samples

Leach Test	Flotation Test No.	Leach Feed Grade			Extraction %			Leach Residue Grade			ATS Consumption (kg/mt)
		Au (g/mt)	Ag (g/mt)	% Cu	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	
27	17	0.08	11.3	0.21	89.7	68.5	51.0	0.06	4.3	0.195	51.6
37	20	0.12	10.0	0.43	69.8	54.7	24.8	0.11	<7	0.326	50.2
38	21	0.49	7.1	0.31	36.6	41.6	25.1	0.47	<7	0.239	48.8
39	22	0.38	13.0	0.14	49.5	34.1	13.0	0.35	11.0	0.121	52.0
40	23	0.07	8.5	0.29	77.2	28.4	7.1	0.08	<7	0.259	54.5
41	24	0.10	40.0	0.35	78.1	15.2	21.0	0.09	36.6	0.320	45.9
42	25	0.17	10.0	0.44	65.8	20.7	17.5	0.16	9.3	0.343	44.6
43	26	0.10	5.0	0.22	74.1	27.9	14.7	0.10	<7	0.195	50.5
44	27	0.07	8.0	0.12	34.6	18.4	20.7	0.49	<7	0.097	39.0
46	LCT1-T	-	-	-	72.7	28.3	33.6	0.10	16.6	0.206	54.3
48	LCT2-T	-	-	-	70.8	22.6	13.3	0.12	20.0	0.659	62.0

These results indicated the following:

- Majority of the gold and some silver and copper are extracted using ATS leach.
- The ammonium hydroxide and ammonium thiosulfate consumptions were high. For example, approximately 100 kg/mt of ammonium hydroxide and 30 to 55 kg/mt of ATS were consumed to extract the precious metals.
- The calculated head for gold was significantly higher than assayed head (± 0.5 g/t vs. 0.15 g/t Au). Hence, the gold extraction based on feed and residue assays were lower than shown in the tables. This is due to interference of ammonium hydroxide in the analytical procedures. The recoveries were calculated based on feed and residue assays. The biggest difference was for gold. Approximately 15% lower recovery was calculated as compared to recovery based on solution and residue assays.
- Locked-cycle testing of recycling ATS demonstrated that it has no effect on leach kinetics up to six cycles (Table 13-9).
- ATS testing of variability samples clearly demonstrate that the extractions based on solution analyses and residue are erroneous (Table 13-10). Leach test No. 44 had feed assaying 0.07 g/t Au and the residue assaying 0.49 g/t Au.

13.2.5 Sulfuric Acid Leaching

The mineralogy of the composite samples from earlier test work has indicated that copper minerals present in the ore are secondary copper (chalcocite/covellite), oxide copper, and copper silicates. The flotation process can recover secondary copper and some oxide copper following the sulfidization process but will not recover copper silicates. These are readily leachable with sulfuric acid.

Flotation tailing from varying grinds, variability samples, and locked-cycle tests were subjected to sulfuric acid leaching. The acid concentration was maintained at 20 g/L. The tests were originally run for 24 hours but later tests were performed for only 8 hours since most of the copper is extracted in ± 6 hours.

The test results are summarized in Table 13-11 and Table 13-12.

Table 13-11: Summary of Acid Leach Test on Flotation Tailings at Varying Grind Sizes

Test No.	Flotation Test No.	Grind P_{80} Mesh	Extraction % Cu		Leach Residue % Cu	Calculated Head % Cu	Acid Consumption (kg/mt)	
			8 Hrs	24 Hrs			8 Hrs	24 Hrs
Endo/Exo MC								
15	9	65	72.4	78.9	0.059	0.2799	53.47	71.41
16	10	100	77.3	80.5	0.049	0.2526	55.18	72.93
17	11	150	80.4	80.7	0.050	0.2563	57.32	75.21
18	12	200	72.6	82.3	0.049	0.2763	61.11	79.59
Magnetite MC								
19	13	65	-	55.5	0.2576	0.5793	-	38.93
20	14	100	51.6	58.9	0.2664	0.6487	31.10	38.87
21	15	150	60.2	58	0.2703	0.6441	30.84	38.24
22	16	200	56.1	63.8	0.2775	0.7660	36.74	45.17

Note: Sulfuric Acid addition was 20 g/L

Table 13-12: Summary of Acid Leach Test on Flotation Tailings for Variability Samples

Test No.	Tailing Flotation Test	Ore Type	Extraction % Cu		Calculated Head % Cu	Acid Consumption (kg/mt)	
			8 Hrs	24 Hrs		8 Hrs	24 Hrs
28	17	Endo/Exo	80.8	82.2	0.308	57.30	74.85
29	20	Exo	75.6	-	0.468	37.96	-
30	21	Exo	76.5	-	0.3606	20.97	-
31	22	Endo	39.3	-	0.152	39.3	-
32	23	Endo	43.1	-	0.320	25.1	-
33	24	Endo	64.4	-	0.466	51.8	-
34	25	Endo/Exo	74.9	-	0.541	42.8	-
35	26	Endo	62.4	-	0.295	27.0	-
36	27	Mag HG	45.2	-	0.123	14.56	-
45	LCT-1	Endo/Exo MC	84.3	-	0.392	61.2	-
46	LCT-2	Mag MC	62.3	-	0.789	36.5	-

Note: Sulfuric Acid addition was 20 g/L

These results indicated the following:

- A majority of the copper in the Endo/Exo MC flotation tailing is leached in 8 hours. The leached residue assayed ± 0.05% Cu. The acid consumption was ± 75 kg/mt.
- The extraction of copper from Magnetite MC flotation tailing was low (± 55%). However, the acid consumption was ± 40 kg/mt.
- The variability leach results indicate that 8 hours of leach consumes significantly less acid (57.3 kg/t) as compared to 24 hours of leach (75 kg/t). The additional copper extraction was only 1.4%.

These results indicate that sulfuric acid will extract copper which did not float (i.e. copper silicates).

13.2.6 Flotation Testing

Rougher flotation tests were performed with the two composites to determine the amenability of recovering gold, silver, and copper. The tests were performed at varying grind sizes with and without sulfidization of the oxide minerals. The reagent suite consisted of potassium amyl xanthate, Aero promotors 208 and 404, and methyl isobutyl carbinol (MIBC) as a frother. NaHS was used as a sulfidizing agent.

The results are summarized in Table 13-13 and Table 13-14.

Table 13-13: Summary of Flotation Test Results for Endo/Exo and Magnetite Master Composites for Varying Grind Sizes

Test No.	Grind P_{80} Mesh	Recovery %				Concentrate Grade			Calculated Feed		
		Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
Endo/Exo Composite											
1	65	2.6	40.4	28.6	21.7	2.24	196.4	3.50	0.15	18.1	0.43
2	100	4	45.3	36.2	23.0	1.58	156.0	2.46	0.14	17.3	0.43
3	150	3.6	44.3	32.1	24.9	1.53	188.1	2.99	0.12	21.4	0.44
4	200	5.8	35.6	26.9	21.2	1.04	90.3	1.50	0.14	19.5	0.41
Magnetite Composite											
5	65	10.2	32.6	12.4	12.6	0.49	19.8	0.80	0.14	16.4	0.65
6	100	7.6	38.6	22.3	9.2	0.59	31.0	0.82	0.14	10.6	0.68
7	150	11.1	42.5	21.9	12.4	0.46	30.0	0.77	0.13	15.1	0.69
8	200	14.5	47.3	25.5	17.4	0.39	26.1	0.86	0.13	14.8	0.72

Note: Flotation Time: 9 Minutes

Table 13-14: Summary of Flotation Test Results for Endo/Exo and Magnetite Master Composites for Varying Grind Sizes and Sulfidization

Test No.	Grind P_{80} Mesh	Recovery %				Concentrate Grade			Calculated Feed		
		Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
Endo/Exo Composite											
9	65	3.6	47.3	60.5	45.6	1.67	280.4	5.91	0.13	16.8	0.47
10	100	2.9	51.8	38.5	45.4	2.48	393.8	7.04	0.14	30.1	0.46
11	150	2.9	44.0	66.3	46.4	2.11	422.0	7.73	0.14	18.4	0.48
12	200	3.3	47.1	76.2	44.6	2.07	429.8	6.0	0.15	18.8	0.45
Magnetite Composite											
13	65	7.5	41.4	27.0	14.0	0.88	62.9	1.25	0.16	17.4	0.67
14	100	8.8	38.5	30.4	15.7	0.65	42.5	1.20	0.15	12.3	0.67
15	150	15.4	47.3	48.1	24.6	0.39	38.2	1.05	0.13	12.2	0.66
16	200	11.1	47.1	42.9	19.9	0.64	48.0	1.16	0.15	12.4	0.65

Note: Flotation Time: 9 Minutes
Sulfidization with 500 g/mt Natts

These results indicated the following:

- The rougher concentrate recovered 35% to 45% of gold, 26% to 36% of silver, and 21% to 25% of copper from Endo/Exo MC. The concentrate assayed 1 to 2.2 g/t Au, 90 to 196 g/t Ag, and 1.5% to 3.5% Cu.
- The rougher concentrate for the Magnetite MC recovered 33% to 47% of gold, 12% to 25% of silver, and 9% to 17% of copper. The concentrate grades were lower because a significantly higher proportion of feed floated (i.e. ± 10% as compared to ± 4% for Endo/Exo MC).
- Sulfidization of the oxide minerals significantly improved both recovery and grade of silver and copper in the rougher concentrate. The concentrate from Endo/Exo MC assayed 1.67 to 2.48 g/t Au, 280 to 430 g/t Ag, and 6% to 7.7% Cu. The metal recoveries in the concentrate were 44% to 52% of gold, 38.5% to 76.2% of silver, and ± 46% of copper.

Two 10-kg rougher flotation tests were performed to generate a concentrate for cleaner flotation tests with the primary objective of determining if marketable-grade copper concentrate can be produced. The rougher flotation results are summarized in Table 13-15, and the cleaner flotation results using the process flowsheet shown in Figure 13-3 are summarized in Table 13-16.

Table 13-15: Rougher Flotation Test at P_{80} of 100 Mesh in One Cubic Foot Flotation Cell for Endo/Exo Master Composite (Test No. 17)

Product	Distribution %				Assays		
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu
Rougher Concentrate	1.3	51.4	46.9	39.3	6.05	762	10.47
Rougher Tailing	98.7	48.6	53.1	60.7	0.08	11.3	0.21
Calculated Feed	100	100	100	100	0.15	21.0	0.345

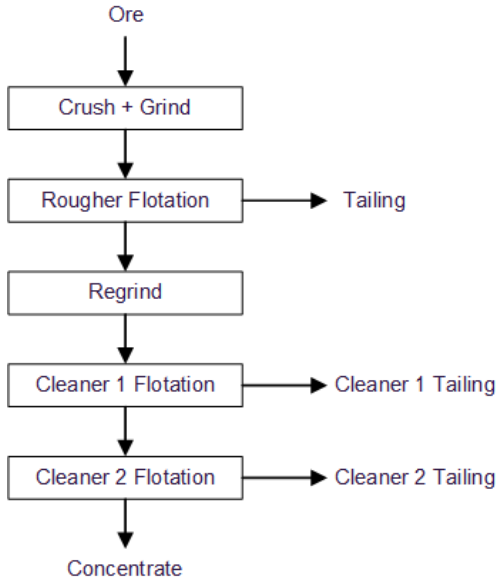


Figure 13-3: Rougher and Cleaner Open-Circuit Test Flowsheet

Table 13-16: Cleaner Flotation Tests With/Without Regrind of Rougher Concentrate from Flotation Test No. 17

Product	Distribution %				Assays		
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu
Test No. 18 (No Regrind)							
Cleaner 2 Conc	23.0	53.8	70.0	74.8	16.0	2330	34.14
Cleaner 2 Tail	10.3	7.4	7.5	5.8	4.92	557	5.93
Calc Cleaner 1 Conc	33.4	61.3	77.5	80.7	12.57	1780.6	25.40
Cleaner 1 Tail	66.6	38.7	22.5	19.3	3.98	259	3.05
Calc Rougher Conc	100	100	100	100	6.85	767	10.51
Test No. 19 (2.5 min Regrind)							
Cleaner 2 Conc	16.6	51.0	56.2	62.2	16.10	2560	39.05
Cleaner 2 Tail	7.1	9.0	5.3	4.9	6.63	563	7.22
Calc Cleaner 1 Conc	23.7	60.0	61.5	67.1	13.26	1961.6	29.51
Cleaner 1 Tail	76.3	40.0	38.5	32.9	2.75	382	4.50
Calc Rougher Conc	100	100	100	100	5.24	756	10.43

These results indicated the following:

- One-cubic-foot flotation cell produced similar recoveries as the small flotation cell but recovered a higher grade concentrate. The overall weight recovery was 1.3%.
- The cleaner flotation tests indicated that a marketable-grade copper concentrate can be produced from Endo/Exo MC with one cleaner-flotation stage and no regrind required. The concentrate assayed 12.57 g/mt Au, 1780.6 g/mt Ag, and 25.40% Cu.

13.2.6.1 Locked-Cycle Tests

A locked-cycle flotation test was performed to simulate the flotation process for recovery of copper and precious metals in a plant.

A six-cycle locked-cycle test employed the process flowsheet shown in Figure 13-4. The test results for the two composites are summarized in Table 13-17.

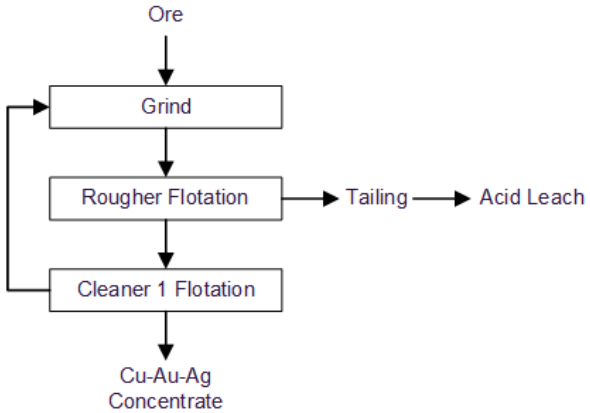


Figure 13-4: **Locked-Cycle Test Flowsheet**

Table 13-17: Summary of Locked-Cycle Test Results

Product	Distribution %				Assays		
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu
Endo/Exo MC							
1st Cl Conc	0.4	39.3	36.2	33.5	17.43	2288	35.7
1st Cl Tail	0.1	0.8	0.5	0.3	2.184	203	1.8
Calc Ro Conc	0.5	40.1	36.7	33.8	14.38	1871	28.92
Ro Tail	99.5	59.9	62.3	66.2	0.11	15.94	0.282
Calc Feed	100.00	100.0	100.0	100.0	0.18	25.3	0.43
Magnetite MC							
1st Cl Conc	0.4	20.9	11.6	3.8	10.68	563	6.0
1st Cl Tail	0.1	0.3	0.2	0.2	0.628	47	1.0
Calc Ro Conc	0.5	21.2	11.8	4.0	8.67	459.8	5.0
Ro Tail	99.5	78.8	88.2	96.0	0.15	16.29	0.572
Calc Feed	100.0	100.0	100.0	100.0	0.19	18.5	0.60

These results indicated the following:

- The cleaner 1 flotation concentrate, assaying 35.7% Cu, 17.43 g/t Au (0.5090 opt), and 2288 g/t Ag (66.81 opt), recovered 33.5% of copper, 39.2% of gold, and 36.2% of silver in 0.4% of the feed weight for Endo/Exo MC.
- The Magnetite MC did not produce a marketable-grade copper product. The cleaner 1 concentrate assayed only 6% Cu.

It is important to note that one will not process 100% Magnetite material in the plant. Additional locked-cycle tests should be run with a blend of Endo/Exo and Magnetite material based on the mine plan.

13.2.6.2 Open-Cycle Rougher and Cleaner Flotation Tests for Variable Feed Materials

Open-cycle rougher and cleaner flotation tests were performed with the variability samples. The test results are summarized in Table 13-18 and Table 13-19.

Table 13-18: Flotation Test Results for Variability Samples

Test No.	Ore Type	Sample Description	Rougher Recovery %				Tailing Grade			Calculated Feed		
			Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
17	Endo/Exo	MC	1.3	51.4	46.9	39.3	0.08	11.3	0.21	0.15	21.0	0.345
20	Exo	102	3.3	58.6	45.0	11.4	0.12	10.0	0.43	0.35	19.9	0.50
21	Exo	101/103	1.4	50.1	20.3	10.2	0.49	7.1	0.31	0.97	8.8	0.34
22	Endo	298/667	4.9	62.6	33.4	70.6	0.38	13.0	0.14	0.97	18.5	0.47
23	Endo	104/596	1.8	25.7	3.5	4.8	0.07	8.5	0.29	0.093	8.6	0.30
24	Endo	291	4.6	14.8	4.3	7.3	0.10	40.0	0.35	0.11	40.0	0.36
25	Endo/Exo	679/14	5.1	48.1	31.7	12.9	0.17	10.0	0.44	0.31	13.9	0.48
26	Endo	127/128	1.3	45.3	12.6	13.7	0.10	5.0	0.22	0.18	5.6	0.25
27	Mag	High Grade	0.7	73.8	8.1	3.9	0.74	8.0	0.12	2.81	8.6	0.12

Note: 9 minutes of rougher flotation at P_{80} = 100 mesh

Table 13-19: Cleaner Concentrate Grades for Variability Samples

Test No.	Ore Type	Sample Description	Calculated Feed Grade			1st Cleaner Concentrate Grade		
			Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
17	Endo/Exo	MC	0.15	21.0	0.345	-	-	-
20	Exo	102	0.35	19.9	0.50	17.9	802	2.04
21	Exo	101/103	0.97	8.8	0.34	13.7	381	6.22
22	Endo	298/667	0.97	18.5	0.47	31.8	307	17.43
23	Endo	104/596	0.093	8.6	0.30	4.01	25.0	1.07
24	Endo	291	0.11	40.0	0.36	0.76	67.0	0.58
25	Endo/Exo	679/14	0.31	13.9	0.48	14.2	398	2.81
26	Endo	127/128	0.18	5.6	0.25	18.4	145	6.95
27	Mag	High Grade	2.81	8.6	0.12	1129	350	1.61

Note: 9 minutes of rougher flotation at P_{80} = 100 mesh
6 minutes of cleaner flotation

These results indicated the following:

- There appears to be no correlation between copper recovery in the flotation process and feed grade. However, the overall recovery of copper in the combined flotation and leach process is over 90% (Figure 13-5). A few samples did show lower copper recovery.
- The grade of the cleaner 1 concentrate (Table 13-19) indicated three of eight samples had the potential to produce ± 25% copper concentrate. The grades for cleaner 1 ranged from 6.22 to 17.43% Cu. It appears that two cleaner stages may be needed to achieve ± 25% copper in the concentrate.
- Both copper recovery and concentrate grade for the Magnetite composite and high-grade Magnetite composite were not good.
- The higher the gold feed grade, the higher the gold recovery in the flotation process (Figure 13-6).
- The silver in the copper concentrate will help in marketing lower grade copper concentrate.

More flotation testing is recommended to ensure that the flotation concentrate can be marketed.

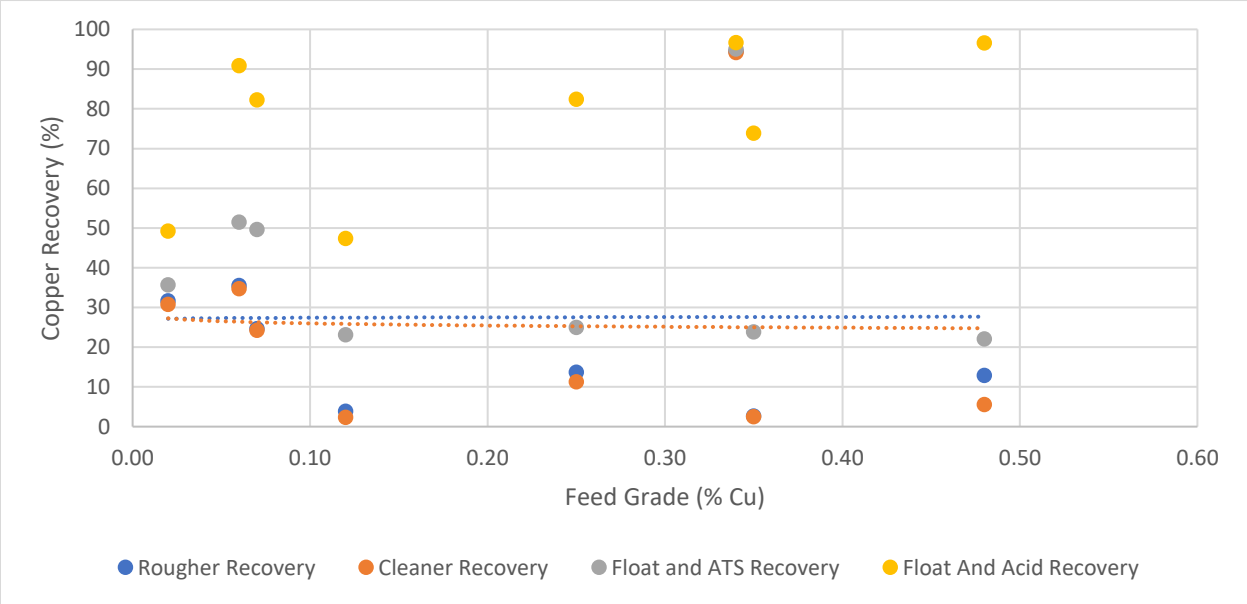


Figure 13-5: Copper Recovery as Function of Feed Grade

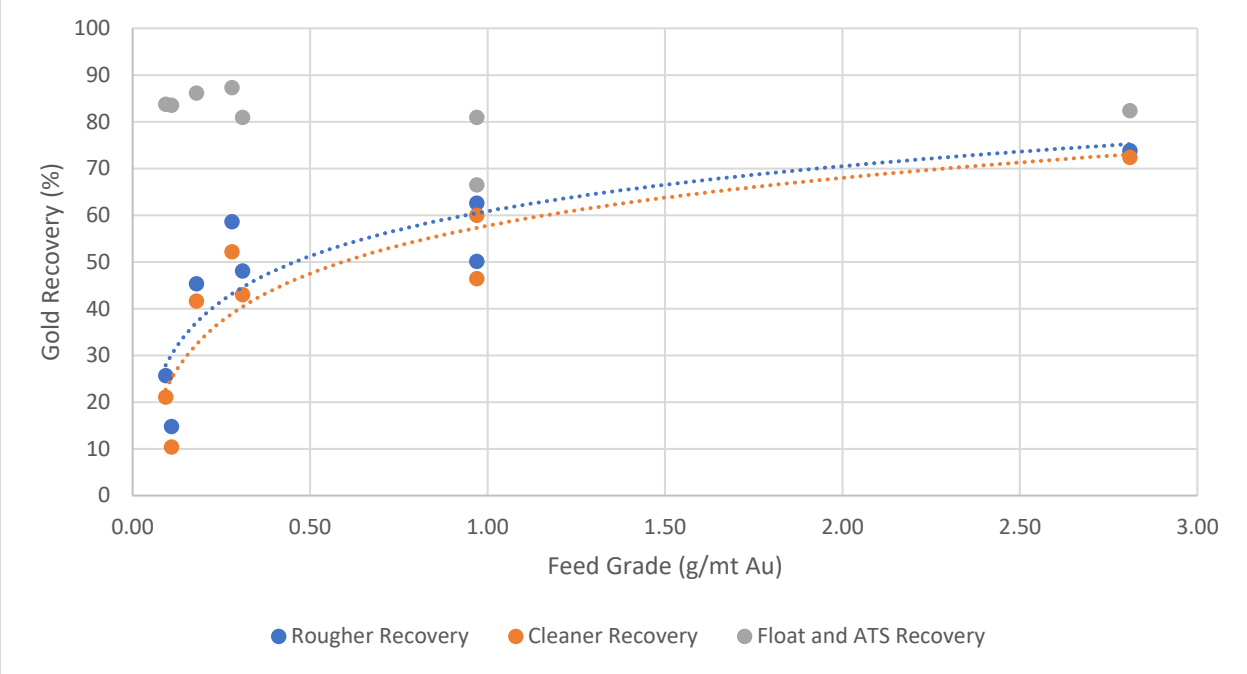


Figure 13-6: Gold Recovery as Function of Feed Grade

13.2.7 Smelter Penalty Analyses of Flotation Concentrates

The cleaner 1 concentrates from the two locked-cycle tests were submitted for smelter penalty analyses. The concentrate from the Endo/Exoskarn master composite assayed 35.3% Cu (LCT-1), whereas the concentrate from the Magnetite master composite assayed only 6% Cu (LCT-2). The smelter penalty and chemical analyses of the concentrate are summarized in Table 13-20 and Table 13-21.

Table 13-20: Smelter Penalty Analyses of Copper Concentrates

Element	Endo/Exo Conc (LCT-1)	Magnetite Conc (LCT-2)
As, ppm	2937	841
Bi, ppm	3544	398
Sb, ppm	47	138
Hg, ppm	16.70	66.10
Se, ppm	298	33
Te, ppm	388	88
SiO_2, %	20.07	41.29
MgO, %	2.25	2.00
P_2O_5, %	0.41	0.83
Chloride, ppm	19,600	104
Fluoride, ppm	204	500

Table 13-21: Chemical Penalty Analyses of Copper Concentrates

Element %	Copper Concentrate	
	Endo/Exo MC (LCT-1)	Magnetite MC (LCT-2)
Al	0.93	4.70
Ca	2.28	3.01
Cu	34.61	6.52
Fe	13.81	16.61
K	0.32	0.56
Mg	1.30	1.21
Mn	0.17	1.31
Mo	0.16	0.07
Na	0.16	0.07
Pb	0.26	0.11
Ti	0.04	0.20
Zn	0.40	0.83
ppm		
Ag	1936	514
Ba	244	751
Cd	43	33
Co	132	140
Cr	43	92
Ni	165	109
Sr	48	117
V	37	68
W	497	0.83

The deleterious elements in the concentrate are arsenic, antimony, cadmium, selenium, bismuth, and tellurium. Limits are also imposed on mercury, lead, zinc, and nickel.

Additional testing needs to be undertaken to determine the level of the impurities in the copper concentrate which will assist in marketing the concentrate.

13.2.8 Thickening Test

Thickening tests were performed on flotation tailings for both Endo/Exo and Magnetite master composites. These samples were generated at a primary grind of P_{80} of 100 mesh. A high weight anionic polymer (DAF-10) manufactured by Polymer Ventures was used at 17 g/mt to thicken the solids. Additional tests were also performed following the acid leaching of the flotation tailings. The test results are summarized in Table 13-22.

Table 13-22: Thickening Test Results

Sample	P_{80} µm	pH	Flocculant	Feed % Solids	Unit Area Required (ft^2/t/day)		
					45%	50%	55%
Endo/Exo Flotation Tailing	150	7.9	17 g/t DAF-10	25	0.015	0.038	0.33
Mag MC Flotation Tailing	150	7.9	17 g/t DAF-10	25	0.035	0.089	6.384
Endo/Exo Flotation Tail Acid Leach Residue	150	3.0	17 g/t DAF-10	16	0.799	2.829	-
Mag MC Flotation Tailing Acid Leach Residue	150	3.0	17 g/t DAF-10	15	0.202	0.498	-

These results indicated the following:

- Flotation tailings can be thickened to 55% solids for the Endo/Exo MC using 17 g/mt of flocculant DAF-10 or any other similar flocculant. Unit area required was calculated to be 0.33 ft^2/t/day.
- Magnetite MC flotation tailings would require 6.384 ft^2/t/day to thicken to 55% solids. However, one would not be processing this material separately in the plant.
- The thickener area requirements for acid leach residues were high and the thickener diameter would be very large.

The acid leach residue can be directly sent to belt filter where the pregnant solution can be separated, and the residue can be easily washed.

13.2.9 Filtration Tests

Filtration tests were performed on acid leach residue for both Endo/Exo and Magnetite composites. Both vacuum and pressure filtration tests were performed. The test results are summarized in Table 13-23.

Table 13-23: Summary of Vacuum and Pressure Filtration Results

Sample	Feed % Solids	Vacuum in Hg	Pressure psi	% Cake Moisture	Dry lbs/ft^2/hr
Endo/Exo MC Leach Residue	38	16.5	-	24.5	128.4
Magnetite MC Leach Residue	39	16.5	-	25.9	95.2
Endo/Exo MC Leach Residue	41	-	80	16.2	911.3
Endo/Exo MC Leach Residue	40	-	80	15.7	1103.8
Magnetite MC Leach Residue	39	-	80	16.6	574.7
Magnetite MC Leach Residue	39	-	80	17.2	272.3

Note: Grind P_{80} = 100 mesh - Feed: Flotation Tailing

These results indicated the following:

- Vacuum filtration rates were significantly lower than pressure filtration rates.

Additional studies with a mixture of the two ore types are recommended in order to accurately size the belt filter.

13.2.10 Cementation Testing

A preliminary cementation test was performed on the pregnant solution from acid leaching of Endo/Exo master composite flotation tailings. The results are summarized in Table 13-24 and Table 13-25.

Table 13-24: Solution Analyses of Pre- and Post-Cementation for Endo/Exo Leach Test

Element, ppm	Pre-Cementation (Preg Solution)	Post-Cementation (Barren Solution)
Al	1158	1244
Ca	498	631
Cd	3.58	3.11
Co	8.22	8.98
Cu	1608	90.6
Fe	3542	6211
P	364	25.9

Table 13-25: Analyses of Cemented Copper

Element %	Cemented Copper
Cu	28.7
Fe	41.2
P	2.34
S	1.01
Zn	0.04

The cementation test was performed for a residence time of 2.5 hours. Approximately 94.4% of the copper was precipitated. Due to excess residence time, a significant amount of iron fillings were dissolved. The cemented copper assayed 28.7% Cu and 41.2% Fe.

Additional testing is needed to optimize the process parameters.

13.2.11 Evaluation of Alternative Processing Flowsheets

Due to the mineralogy of the copper minerals in the deposit (i.e. sulfide/oxide/silicates), the low grade of the ore, and the need for non-cyanide lixiviant, several alternative flowsheets were evaluated in order to select the optimum process.

13.2.11.1 Gravity/Flotation Process

The process flowsheet is shown in Figure 13-7. The test results for the gravity and flotation process are summarized in Table 13-26 at 100 mesh for the Endo/Exo and Magnetite master composites. These test results were combined from independent tests performed by gravity and flotation on feed and should be considered as maximum recoveries at these process conditions.

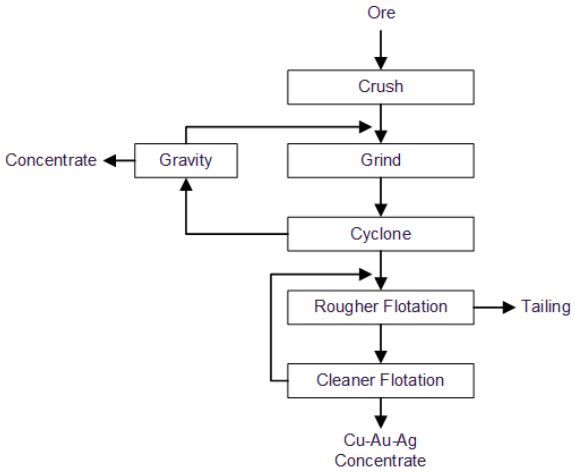


Figure 13-7: Simplified Gravity/Flotation Flowsheet

Table 13-26: Gravity and Flotation Results for the Two Master Composites at P_{80} = 100 mesh

Test No.	Recovery %				Grade		
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu
Endo/Exo MC							
EMGT-2	0.6	10.8	3.9	3.8	2.75	156.0	2.83
EMFT-10	2.9	51.8	38.5	45.4	2.48	393.8	7.04
Combined	3.5	62.6	42.4	49.2	2.53	353.0	6.32
Magnetite MC							
EMGT-6	1.6	6.4	5.4	0.5	0.61	40.6	0.20
EMFT-14	8.8	38.5	30.4	15.7	0.65	42.5	1.20
Combined	10.4	44.9	35.8	16.2	0.64	42.2	1.05

These results indicated the following:

- The gravity concentration process recovered a maximum of 10.8% Au for Endo/Exo MC. The gravity recoveries may be low because of the feed assaying only 0.14 g/t Au.
- The flotation process recovered a reasonable amount of gold (38.5% to 51.8%), silver (30.4% to 38.5%), and copper (45.7% to 45.4%).
- The combined gravity and rougher flotation process recovered 62.6% Au, 42.4% Ag, and 49.2% Cu. The concentrate grade assayed 2.53 g/t Au, 353 g/t Ag, and 6.32% Cu for the Endo/Exo MC.
- The copper recovery for the Magnetite MC was only 16.2% in the combined gravity and flotation concentrate. The concentrate grade was only 1.05% Cu.

13.2.11.2 Agitated ATS and Acid Leach Process

The simplified process flowsheet for the agitated leach circuit is shown in Figure 13-8. The ground ore will be first leached with ATS, followed by separation of pregnant solution from solids. The pregnant solution will be sent to the electrowinning (EW) circuit where gold, silver, and some copper will precipitate. The solids will be leached with sulfuric acid to recover copper.

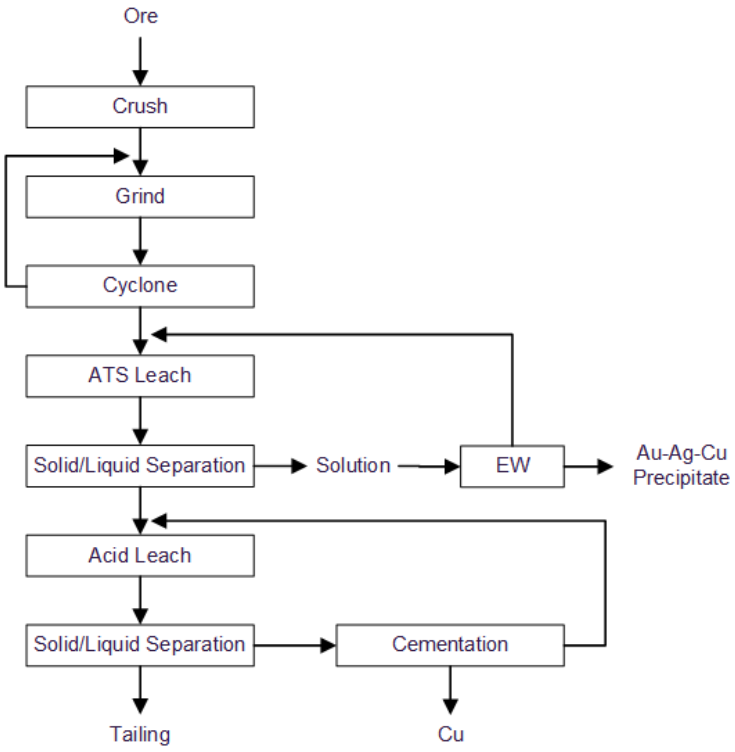


Figure 13-8: Simplified ATS/Acid Leach Flowsheets

Assuming copper that floated was recovered by ATS process, material balance was calculated for both composites at P_{80} of 100 mesh. The results are summarized in Table 13-27.

Table 13-27: Combined Results for Two-Stage Leach Process at P_{80} = 100 mesh

Test No.	Recovery %			Consumption (Reagent kg/t)
	Au	Ag	Cu	
Endo/Exo MC				
EMBR-2	78.6	86.1	43	ATS 42.1
EMBR-16	-	-	41	H_2SO_4 55.0
Total	78.6	86.1	84	
Magnetite MC				
EMBR-6	79.3	46.2	20.5	ATS 49.5
EMBR-20	-	-	41	H_2SO_4 31.1
Total	79.3	46.2	61.5	

Note: 8 hrs leach time for acid

The results indicated the following:

- The recovery of gold, silver, and copper in the two-stage leach process, excluding losses in electrowinning circuit, was 78.6% Au, 86.1% Ag, and 84% Cu for Endo/Exo MC. The recovery from Magnetite MC was 79.3% Au, 46.2% Ag, and 61.5% Cu.

- The reagent consumptions (ATS and acid) were high (31 kg/t to 55 kg/t). Based on the price of $270/mt for ATS, the reagent cost would be $13.50/mt for ATS. Assuming the feed assays 0.5 g/t Au, the incremental recovery using ATS versus flotation would be ± 25% (or △R = $7.50 based on 60 g/t Au).

- The acid consumption of 50 kg/t based on a price of $150/mt equates to $7.50. The incremental copper recovery from flotation tailing would be 55% (90% - 35% based on the locked-cycle test). This is equivalent to $19.25/mt based on $3.50/lb of copper.

These results indicate that the flotation tailing should be subjected to acid leach for recovery of copper instead of ATS leach for recovery of gold.

The assay-by-size test data for flotation tailing is given in Table 13-28. The data indicated that both gold and copper were equally present in all size fractions. Hence, one could not reduce the tonnage to the leach circuit.

13.2.11.3 Flotation/Acid Leach of Tailing

The simplified flowsheet for flotation followed by leaching of the flotation tailing is shown in Figure 13-9. Both ATS and acid leach tests were performed on the flotation tailing. However, acid leaching to recover copper was more economical than recovery of gold. Hence, the flowsheet shows only acid leaching.

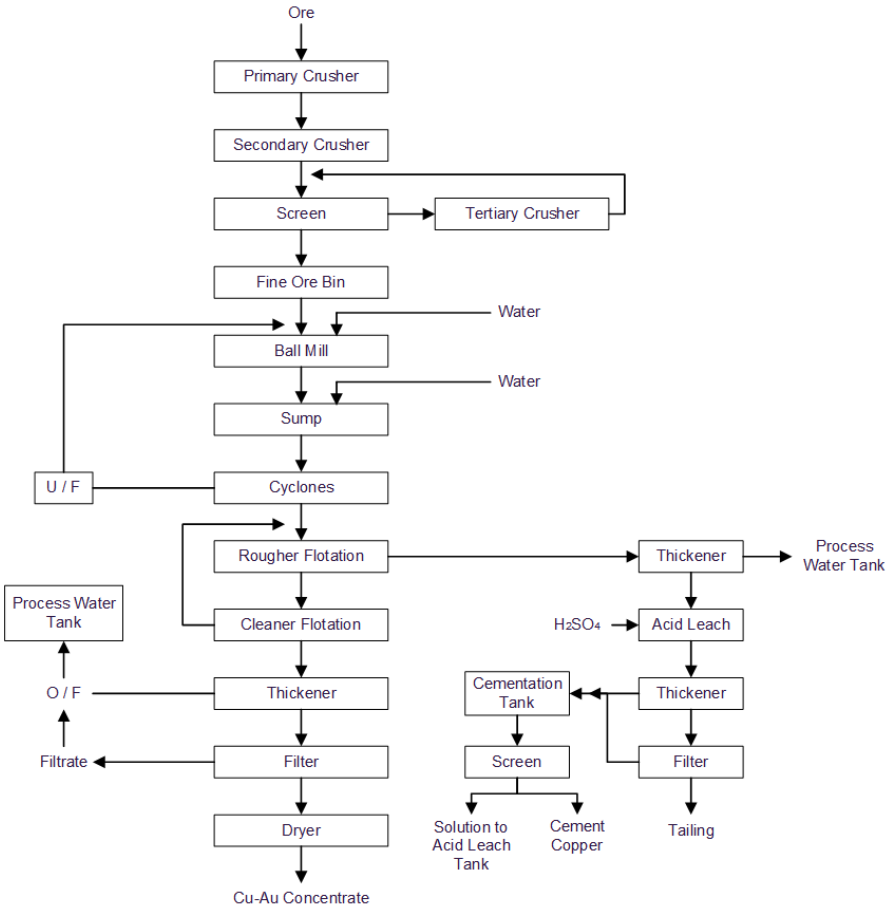


Figure 13-9: Simplified Flotation/Acid Leach Process Flowsheet

The material balance for the flowsheet is presented in Table 13-28. The balance is based on one locked-cycle flotation test on Endo/Exo MC and an assumption that the feed grade would be 0.5 g/t Au, 18.7 g/t Ag, and 0.5% Cu.

Table 13-28: Metal Recoveries for Flotation/Acid Leach Process for Endo/Exo Master Composite

Test No.	Recovery %				Concentrate Grade			Acid Consumption (kg/mt)
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	
LCT-1 Cl1 Conc	0.4	50.0	36.2	33.5	17.43	2288	35.7	
45 Acid Leach	-	-	-	56.1	-	-	-	61.2
Total	0.4	50.0	36.2	89.6	-	-	-	

The overall recoveries of gold, silver, and copper would be 50% Au, 36.2% Ag, and 89.6% Cu.

13.2.12 Conclusions

The following conclusions were drawn based on the metallurgical test work:

- Endoskarn and Exoskarn represent the majority of the oxide ore in the prospect. Magnetite constitutes a small portion of the prospect.
- The metallurgical study was undertaken on two master composites – Endo/Exoskarn and Magnetite ore.
- The Endo/Exo composite assayed 0.14 g/t Au, 18.7 g/t Ag, and 0.43% Cu. The Magnetite composite assayed 0.145 g/t Au, 14.3 g/t Ag, and 0.75% Cu.
- Both composites were slightly abrasive and exhibited average hardness.
- Both composites were not amenable to gravity concentration due to low-grade.
- A marketable-grade copper concentrate could not be produced from the Magnetite composite.
- The copper recovery in the acid leaching of magnetite ore was also low as compared to the Endo/Exo composite.
- Several processing options were evaluated, including gravity/flotation, two-stage leach (ATS and acid), and flotation/acid leach processes. The latter process was the most techno-economically viable.

The full report can be found in Appendix C.

13.3 Forte Analytical – Phase II Testing (2024)

A process flowsheet which produced copper/gold concentrate and cemented copper was developed in Phase I test program as discussed in the previous section 13.2.

Phase II test work was initiated earlier in 2024 with the objective of verifying and/or modifying the process to treat ores in the first several years of operation. Though the test work is ongoing, the available results are summarized in this section.

13.3.1 Preparation of Composites Representing Ores Processed Yearly in the Plant

Composites were prepared based on the yearly mine plan and drill hole material available. The composites represented Years 1, 2, 4, and 4-9. The head analyses of the composites are given in Table 13-9.

Table 13-29: Yearly Composite Sample Head Analyses

Composite Year	Copper %	Sequential Copper % of Total Extracted			Gold troz/st	Hot Shake CN extracted	Silver troz/st	Hot Shake CN extracted	Carbonate %	Sulfide
		Step 1 H2SO4 + Ferric	Step 2 CN	Step 3 Four Acid		Gold troz/st		Silver troz/st		Sulfur %
1	1.28	81%	6%	13%	0.008	0.0038	1.20	0.074	0.35	0.08
2	0.56	92%	1%	7%	0.013	0.0044	0.52	0.104	3.5	0.04
3	0.55	84%	1%	15%	0.012	0.0061	0.41	0.145	2.6	0.03
4-9	0.37	69%	2%	29%	0.05	0.0038	0.50	0.212	6.05	0.04

The results indicated the following:

- The ore for the first three years will assay 1.28% Cu, 0.56% Cu, 0.55% Cu; 0.008 opt Au, 0.013 opt Au, 0.012 opt Au; and 1.20 opt Ag, 0.52 opt Ag, 0.41 opt Ag, respectively.
- The carbonate content varied from 0.35% in Year 1 to 6.05% for Years 4-9.
- The sulfide sulfur content is low in the deposit (0.03% to 0.08%).

13.3.2 Rougher Flotation Tests

Rougher flotation tests were performed with the yearly composites at three different grinds using the same reagent suite established in Phase I of the test program. The results are summarized in Table 13-30.

Table 13-30: Rougher Flotation Results for Varying Grind Sizes

Year Comp	Grind P80 Mesh	Recovery %				Grade		
		Wt	Cu	Au	Ag	% Cu	Au (g/t)	Ag (g/t)
1	100	4.78	33.5	50.5	54.8	8.6	4.08	518
1	150	4.97	30.8	47.1	53.8	7.5	3.38	471
1	200	4.57	33.6	43.0	53.8	9.1	2.95	528
2	100	3.06	24.3	49.9	32.4	5.8	10.02	252
2	150	3.92	21.7	39.2	30.7	3.5	4.50	172
2	200	2.86	21.3	50.1	41.8	5.1	-	-
3	100	3.52	12.0	24.9	14.5	1.9	3.27	74
3	150	3.96	16.4	44.6	33.7	2.3	4.86	121
3	200	3.75	15.6	43.5	21.0	2.3	4.31	113
4-9	100	5.81	22.4	35.7	28.1	1.4	0.95	74
4-9	150	6.35	27.0	47.0	56.7	1.3	1.12	73
4-9	200	6.56	25.6	46.6	27.6	1.2	1.40	54

Flotation Time: 9 Minutes

The results indicated the following:

- Primary grind of P80 of 100 mesh was optimum for Year 1 and 2 composites, whereas, P80 of 150 mesh was optimum for Years 3 and 4-9.
- The flotation concentrate recovered 45% to 50% of gold, 33% to 55% of silver, and 16% to 33% of copper.

These results were similar to those obtained in Phase I of the test program.

13.3.3 Open-Cycle Rougher-Cleaner Flotation Test

An open-cycle rougher-cleaner flotation test was performed to determine if one could produce marketable-grade copper concentrate. The test results for the four composites are presented in Table 13-31 to Table 13-34. The material balances for the composites could not be obtained because of insufficient sample for analyses. This will be undertaken when 10 kg tests are performed.

Table 13-31: Flotation Mass Balance for Year 1 Composite Sample

Year 1 Composite	Mass (g)	Indiv Mass	Cum Mass	Copper			Gold		Silver	
Material		Fraction	Fraction	Grade %	g	Ind Dist	oz/st	Ind Dist	troz/st	Ind Dist
Feed (measured)	2000			1.2%			0.011		1.20	
Feed (calculated)	2000			1.3%	25.98					
Rougher Con (Calc)	88.8	4.44%		9.46%	8.40	32.3%	0.080	33%	15.0	56%
Cleaner 1 Con (Calc)	27.5	1.38%		**23.25%**	6.39	**24.6%**	**0.207**	**26%**	**38.1**	**44%**
Cleaner 1 Tail	61.3	3.07%	3.07%	3.27%	2.00	7.7%	0.023	6%	4.64	12%
Cleaner 2 Con (calc)	14.2	0.71%	3.78%	**35.9%**	5.10	**19.6%**	**0.222**	**15%**	**55.6**	**33%**
Cleaner 2 Tails	13.3	0.67%	4.44%	9.74%	1.30	5.0%	0.192	12%	19.5	11%
Comb Cleaner Tail (Calc)	74.6	3.73%		4.42%	3.30	12.7%	0.053	18%	7.28	23%
Rougher Tails	1911.2	95.56%	100.0%	0.92%	17.58	67.7%	0.008	67%	0.55	44%

Table 13-32: Flotation Mass Balance for Year 2 Composite Sample

Year 2 Composite	Mass (g)	Indiv Mass	Cum Mass	Copper			Gold		Silver	
Material		Fraction	Fraction	Grade %	g	Ind Dist	oz/st	Ind Dist	troz/st	Ind Dist
Feed (measured)	2000			0.683%			0.013		0.525	
Feed (calculated)	2000			0.674%	13.48				0.668	
Rougher Con (Calc)	80.6	4.03%		4.58%	3.69	27.4%	0.138	43%	6.85	41%
Cleaner 1 Con (Calc)	11.5	0.58%		**15.54%**	1.79	**13.3%**	**0.362**	**16%**	29.6	32%
Cleaner 1 Tail	69.1	3.46%	3.46%	2.76%	1.91	14.1%	0.101	27%	3.07	20%
Cleaner 2 Con	5.4	0.27%	3.73%	**26.2%**	1.41	**10.5%**	**0.391**	**8%**	53.6	28%
Cleaner 2 Tails	6.1	0.31%	4.03%	6.11%	0.37	2.8%	0.335	8%	8.32	5%
Comb Cleaner Tail (Calc)	75.2	3.76%		3.03%	2.28	16.9%	0.120	35%	3.49	25%
Rougher Tails	1919.4	95.97%	100.0%	0.51%	9.79	72.6%	0.008	57%	0.408	59%

Table 13-33: Flotation Mass Balance for Year 3 Composite Sample

Year 3 Composite	Mass (g)	Indiv Mass	Cum Mass	Copper			Gold		Silver	
Material		Fraction	Fraction	Grade %	g	Ind Dist	oz/st	Ind Dist	troz/st	Ind Dist
Feed (measured)	2000			0.683%			0.012		0.41	
Feed (calculated)	2000			0.576%	11.51		0.012		0.53	
Rougher Con (Calc)	69.8	3.49%		2.95%	2.06	17.9%	0.117	35%	4.0	26.0%
Cleaner 1 Con (Calc)	7.9	0.40%		**12.64%**	1.00	**8.7%**	**0.758**	**26%**	25.1	18.7%
Cleaner 1 Tail	61.9	3.10%	3.10%	1.71%	1.06	9.2%	0.035	9%	1.3	7.3%
Cleaner 2 Con	1.8	0.09%	3.19%	**31.4%**	0.57	**4.9%**	**1.625**	**13%**	74.0	12.5%
Cleaner 2 Tails	6.1	0.31%	3.49%	7.10%	0.43	3.8%	0.502	13%	10.7	6.1%
Comb Cleaner Tail (Calc)	68	3.40%		2.19%	1.49	13.0%	0.077	22%	2.1	13.4%
Rougher Tails	1930.2	96.51%	100.0%	0.49%	9.46	82.1%	0.008	65%	0.41	74.0%

Table 13-34: Flotation Mass Balance for Year 4-9 Composite Sample

Years 4-8 Composite	Mass (g)	Indiv Mass	Cum Mass	Copper			Gold		Silver	
Material		Fraction	Fraction	Grade %	g	Ind Dist	oz/st	Ind Dist	troz/st	Ind Dist
Feed (measured)	2000			0.683%			0.005		0.496	
Feed (calculated)	2000			0.534%	10.67				0.551	
Rougher Con (Calc)	98.2	4.91%		1.38%	1.36	12.7%	0.033	33%	2.192	19.5%
Cleaner 1 Con (Calc)	11.1	0.56%		**6.96%**	0.77	**7.2%**	**0.219**	**25%**	**11.175**	11.2%
Cleaner 1 Tail	87.1	4.36%	4.36%	0.67%	0.58	5.5%	0.009	8%	0.712	5.6%
Cleaner 2 Con	2.6	0.13%	4.49%	**21.3%**	0.55	**5.2%**	**0.716**	**19%**	**36.459**	8.6%
Cleaner 2 Tails	8.5	0.43%	4.91%	2.57%	0.22	2.0%	0.067	6%	3.442	2.7%
Comb Cleaner Tail (Calc)	95.6	4.78%		0.84%	0.80	7.5%	0.015	14%	0.954	8.3%
Rougher Tails	1901.8	95.09%	100.0%	0.49%	9.32	87.3%	0.004	67%	0.467	80.5%

The test results indicated the following:

- Two stages of cleaner flotation are needed for production of marketable-grade copper concentrate (± 25% Cu).
- The copper concentrate assayed 55.6 opt Ag for Year 1, which will contribute significantly to the revenue.

13.3.4 Acid Leach of Flotation Tailing

The flotation tailings were acid leached for recovering copper. The leach data for yearly composites are summarized in Table 13-35.

Table 13-35: Acid Leach Kinetic Data for Flotation Tailing

Time (hr)	% Extraction			
	Year 1	Year 2	Year 3	Year 4-9
2	61.9	53.2	57.8	28.7
4	75.3	82.1	76.3	56.6
6	75.4	79.7	75.8	55.9
7	77.0	82.2	76.8	60.0
8	77.2	83.0	74.9	59.4
Acid Consumption, kg H2SO4/metric ton ore:	57.25	129.07	89.46	133

The test results indicate the following:

- Majority of the copper is leached in 4 hours.
- The acid consumption is dependent on the amount of carbonate in the sample. The acid consumption varied from 57 kg/t to 133 kg/t.
- The acid consumption as a function of carbonate in the ore is given in Figure 13-10. The lower the carbonate in the ore, the lower the acid consumption.

Based on these results, recovery of oxide copper by acid leaching would be uneconomical. The only way to reduce acid consumption was to float carbonates following the copper/gold/silver flotation.

13.3.5 Carbonate Flotation and Acid Leach

Preliminary tests in the ongoing test program floated 5.5% to 15.2% of the material in the carbonate concentrate. The tailing following the carbonate float were acid leached for copper.

The preliminary test results are given in Table 13-36. The acid consumption was reduced by 31% to 69% for the first three yearly composites.

The ongoing work will optimize the carbonate flotation and leaching of the tailings. The process flowsheet has been modified to include the carbonate flotation. The modified flowsheet has been incorporated into the PFS.

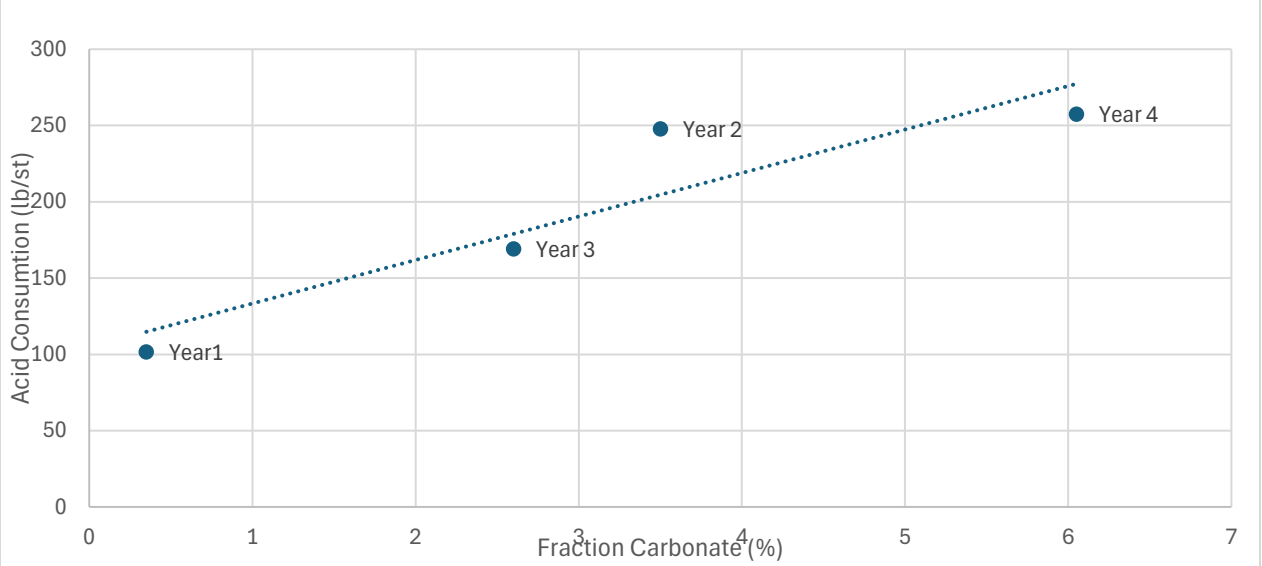


Figure 13-10: Acid Consumption vs Carbonate in Ore

13.3.6 Conclusions

Based on the testwork completed to date, the following conclusions are drawn:

- The flotation process will recover some copper and gold in a marketable-grade copper concentrate.
- The acid leach process will extract the majority of the copper that does not float.
- The combined copper recovery was estimated to be 83%.
- Gold recovery into the copper concentrate was estimated based on historical and current open-circuit rougher-cleaner flotation tests. It is estimated to be 35%.

13.3.7 Recommendations

The following testwork is recommended for the feasibility study:

- Optimize carbonate flotation for the years 1 to 3 samples.
- Optimize the leaching of the carbonate flotation tailings for copper extraction.
- Characterize the combined carbonate concentrate and acid leach residue for environmental/tailings storage facility.
- Prepare and run composites for years 4 to 9 individually through the proposed circuit.

Table 13-36: Comparison of Leaching of Sulfide and Carbonate Flotation Tailing

Test #	Ed's #	Comp Source	% Carbonate	Total Products Mass (g)	Sulfide Float Concentrate Mass (g)	Carbonate Float Concentrate Mass (g)	Carbonate Flot Tails Mass (g)	Fraction Feed Floated	Fraction Orig Sample Floated	6 hr Bottle Roll Leach Preg Cu mg/L	5 hr Agitated Leach Preg Cu mg/L	Apparent Cu lost to Carbonate Conc	Net* Acid Consumed 6 hr kg/t	Net** Acid Consumed 5 hr kg/t	Reduction in Acid Consumed
EMFT 18	FT-1	Y1	0.35	998.26	36.16	54.9	907.2	5.7%	5.5%	5529	3591	35%	50	13	73%
EMFT 19	FT-2	Y2	3.5	998.99	38.59	114.9	845.5	12.0%	11.5%	3301	2227	33%	124	70	43%
EMFT 20	FT-3	Y3	2.6	1012.09	30.89	70.1	911.1	7.1%	6.9%	2802	1498	47%	85	39	55%
EMFT 21	FT-4	Y4-9	6.05	990.38	44.38	150.5	795.5	15.9%	15.2%	1166	529.4	55%	129	94	27%

**Residual Free Acid measured between 15 and 16 g/l in all tests*

***Assumed 15 g/l Residual Free Acid for test 2,3,4 and 8 g/l for test 1*

14. MINERAL RESOURCE ESTIMATES

This section summarizes the mineral resource estimate for the Konnex Project, prepared in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"). The estimate is based on drillhole data and utilizes Ordinary Kriging (OK) implemented with Leapfrog Geo V2023.2.1 software. The reported resources are classified as Measured, Indicated, and Inferred following the "CIM Definition Standards for Mineral Resources and Mineral Reserves", consistent with NI 43-101. Copper, gold, and silver are the primary metals of interest, and all units are imperial with costs reported in US Dollars.

14.1 Data Sources and Compilation

The mineral resource estimate is based on a comprehensive dataset compiled from various sources. Drillhole data represents the primary source of information, encompassing a total of 485 holes (255 historical, 203 recent) drilled with both diamond core (DDH) and reverse circulation (RC) techniques. The total drilled footage is 100,940.8 feet.

In the case of skarn deposits, like the one at the Empire Project, visual observations of mineralization and alteration within the drill core may not be entirely indicative of the original rock types due to the significant geochemical changes caused by the skarnification process. Consequently, skarn zones are treated as distinct rock types within the deposit, rather than altered versions of their original lithology.

To address this and enhance the geological context for the resource estimate, geochemical data is crucial. This data provides a more accurate representation of the current mineralogical and compositional properties of the rock units, which are critical parameters for various applications – including metallurgical performance assessment and environmental impact evaluation – during future resource development.

Therefore, a guided multivariate cluster analysis was employed on the Konnex-owned Empire Project drillhole geochemical data (utilizing both immobile and mobile chemical elements) within ioGAS software. This approach, termed lithogeochemical analysis, leverages the sensitivity of geochemical data to variations in rock types caused by alteration processes. The analysis resulted in a standardized rock unit classification scheme, replacing attempts to define original protolith lithology and alteration signatures separately. This classification scheme improved the accuracy and consistency of rock type identification within the Empire Project database. A ternary classification diagram (Figure 14-1) was subsequently developed based on the most effective discriminating elements (Ca, Fe, and Al/Ti ratio). This diagram facilitates the future classification of geochemical data and offers utility for the rapid classification of new exploration drill core.

Subsequently, the geochemical clusters were compared against the existing logged lithology descriptions. A high degree of concordance was observed between the two datasets, with skarn units identified geochemically corresponding well to logged skarn intervals. However, some discrepancies were noted, particularly near the surface, where weathering processes can modify the geochemistry.

Assay data, specifying the analyzed elements and their detection limits, will be presented in a separate section. Other relevant data sources include density measurements for resource tonnage calculations, topographic surveys for spatial referencing, and historical mine maps (if applicable) to provide insights into past exploration and development activities at the Empire Project.

All data underwent a rigorous verification and cleaning process to ensure accuracy and consistency. This process may have involved identifying and correcting errors, and potentially standardizing data formats for seamless integration.

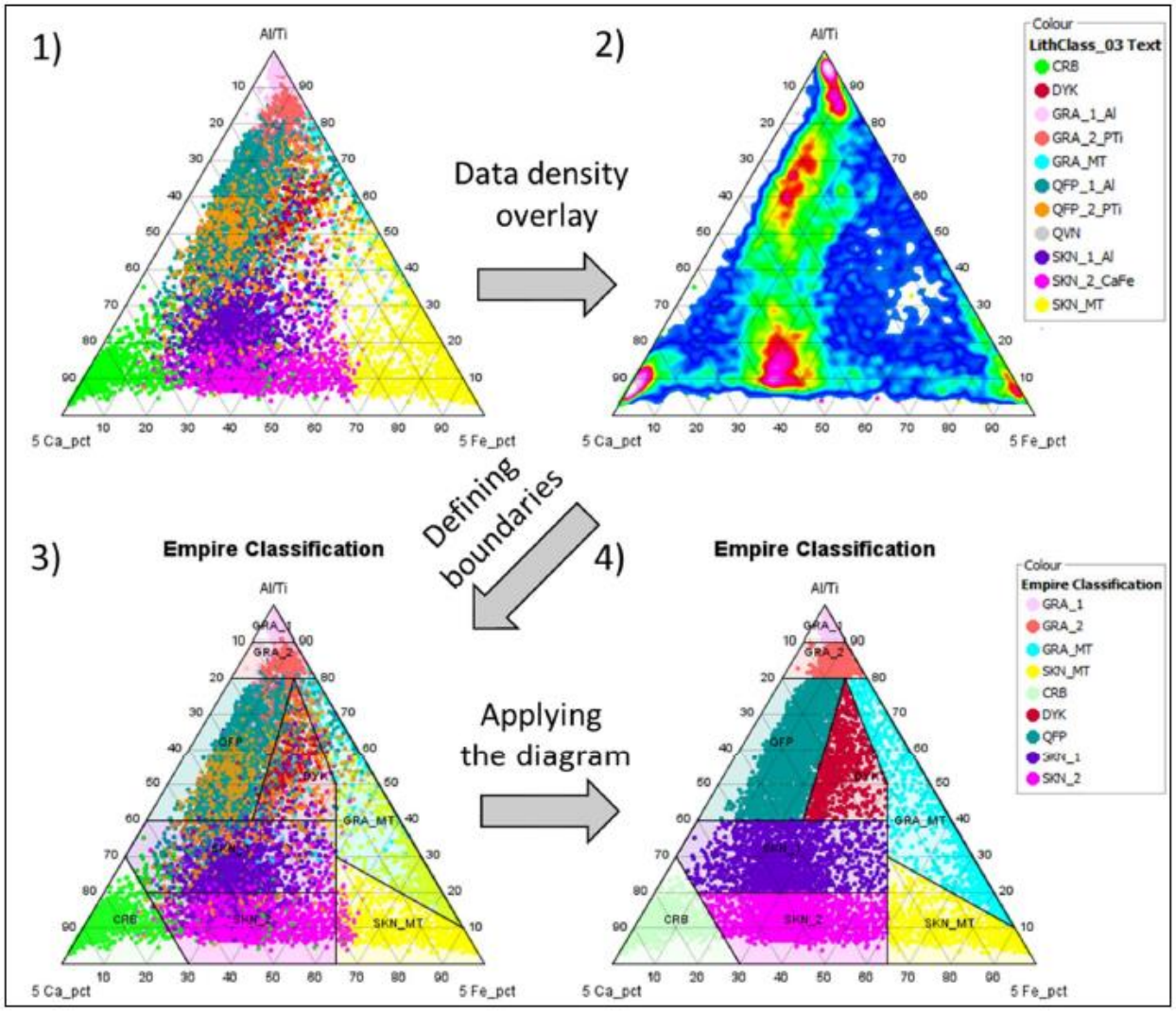


Figure 14-1: Ternary Classification Diagram for Rock Unit Classification Based on Ca, Fe, and Al/Ti ratio.

14.2 Implicit Modeling Method

Following the establishment of the lithogeochemical classification scheme, subsequent data preparation and processing steps were undertaken to construct the implicit geologic model.

The lithology database was filtered to include only drillholes with complete multi-element data. This selection guarantees that geologic wireframes are constructed utilizing drillholes with the most comprehensive geochemical information. Additionally, the lithogeochemical classification was refined for specific categories. Overburden, mine workings, and void intervals were reclassified based on the original drillhole logs, prioritizing field observations for these features. Table 14-1 summarizes the final lithologic codes used in the model.

Table 14-1: Model Lithology Codes

Age	Model Lithology Code	Lithogeochemical Class	Lithology
Oldest	CRB	CRB	Limestone
	GP	QFP_1_Al/QFP_2_PTi	Quartz Feldspar Porphyry (QFP)
	GRA	GRA_1_Al/GRA_2_PTi	Mackay Granite
	SKN-EX	SKN_2_CaFe	Pyroxene Skarn (Exo)
	SKN-EN	SKN_1_Al	Garnet Skarn (Endo)
	SKN-MT	SKN_MT	Magnetite Skarn
	FEBX	GRA_MT	FeOx Breccia
	DYK	DYK	Late Barren Dikes
Youngest	OVB	---	Alluvium/Overburden

Data was then composited into 20-foot majority intervals, aligning with the pre-determined block size and minimum mining width for compatibility with downstream applications. The midpoint of each composite interval was then extracted, resulting in a set of points representing the geologic data throughout the model area.

14.2.1 Fault Blocks

To account for variations in mineralization orientation and to differentiate non-mineralized lithologic units (DYK, GRA, OVB) from mineralized zones, the geologic model was subdivided into eight structural or lithological domains (Figure 14-2).

14.2.1.1 Overburden

The initial step in model construction entailed the extraction of the basal contact of the overburden from drillhole data. These points were then interpolated to create a continuous surface using the existing topographic data. User-defined minimum and maximum offset values of 10 feet and 20 feet, respectively, were incorporated during interpolation to account for potential variations in overburden thickness.

14.2.1.2 Fault Blocks

The model was then separated into 4 additional domains (AP, DYK, Empire, Red Star) based on primary fault zones. The historical underground mine area is separated into north (Empire) and south (AP) fault blocks. The division of these fault blocks is post-mineralization dyke, which intruded along a northeast-trending structure. An additional, unmapped structure further divides the north fault block, separating the North Pit/Red Star fault block from the Empire fault block.

14.2.1.3 Mackay Granite

The granite unit's surface was constructed by selecting composite contact points from drillholes that intersected the granite. These contact points were chosen strategically to represent the boundary where the majority of remaining drillhole intercepts transitioned into the granite unit. This approach effectively defines the granite's outer limits while acknowledging that some scattered granite intercepts may exist beyond this boundary within each domain. For modeling purposes, these outlying granite intercepts are assumed to be misclassified and are reclassified as Mackay Porphyry (GP) in the areas outside the designated granite boundary.

To ensure geological consistency, the final granite surface was refined to align with existing surface geology maps. Additionally, in areas with sparse data, the general strike and dip of the unit were incorporated based on information obtained from underground mine maps.

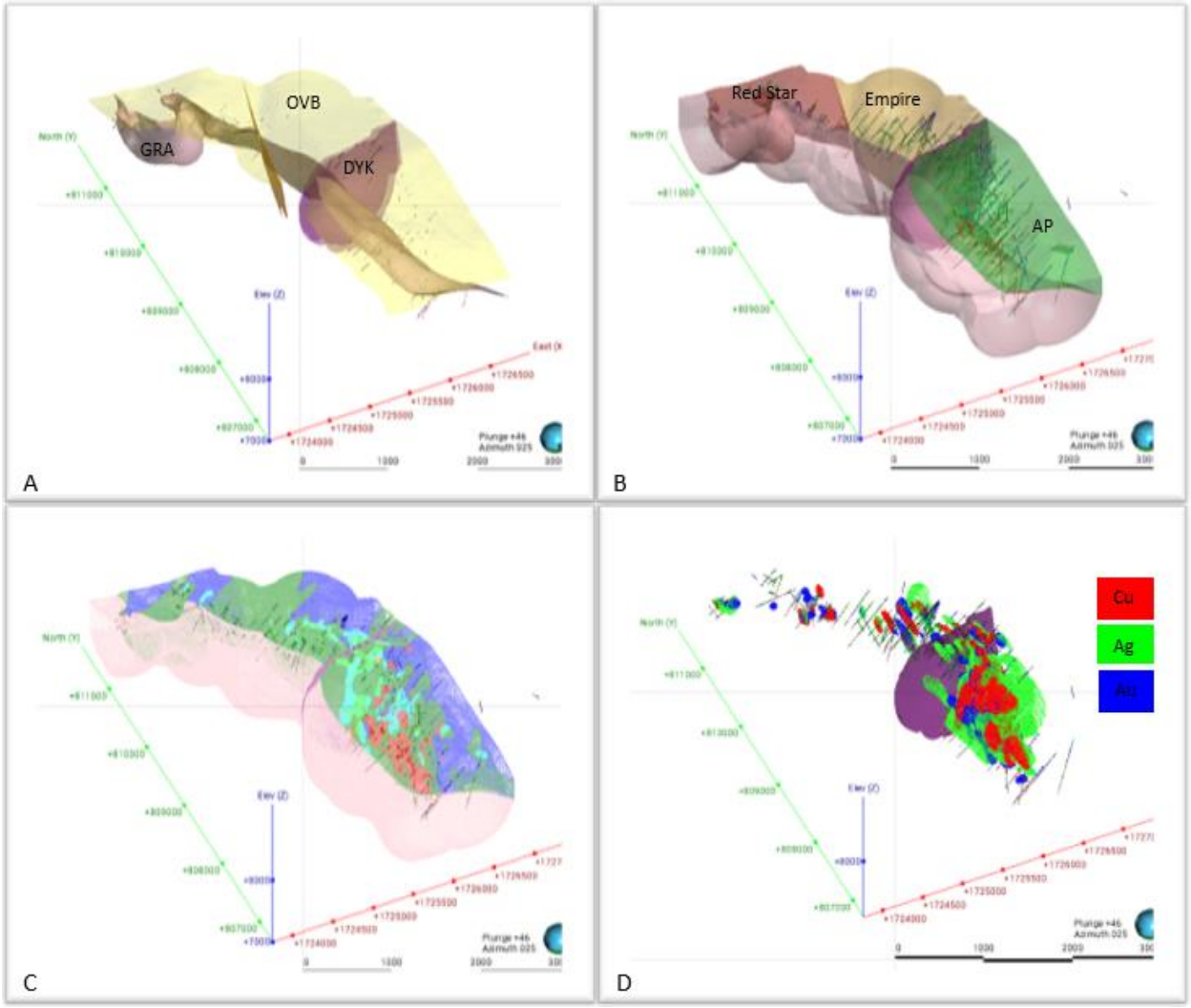


Figure 14-2: Oblique View of the General Workflow to Construct the Geologic Model

(A) Surface models, (B) Fault Block domains (OVB removed for clarity), (C) Combined lithologic model (OVB removed for clarity), (D) Indicator domains (OVB removed for clarity)

14.2.1.4 Indicator Variables and Implicit Modeling

Indicator variables were created for key rock types (SKN_MT, SKN_EN, SKN_EX, FEBX) within the model. These variables assign a value of 1 to points classified as that specific rock type and 0 otherwise. The indicator data was then employed to develop implicit models for each designated rock unit. A threshold value of 0.5 was applied. Points with an indicator value exceeding the threshold were considered to belong to that specific rock type and were incorporated into the corresponding implicit model. This approach defines the spatial extent of each rock unit within the 3D geologic model, resulting in the generation of individual wireframes for each lithology.

During the modeling process, it was identified that some areas logged as SKN_MT or SKN_EN fell outside of the continuous indicator shapes generated for their respective rock types. These specific areas were recoded to the indicator representing a more spatially compatible rock type (FEBX for previously logged SKN_MT and SKN_EX for previously logged SKN_EN).

The GP and limestone (CRB) contact points were used to create a surface splitting the two units. The surface was created using a general trend that followed the regional structural pattern of the dykes with similar characteristics to the GP. The surface was then modified in section and plan view using polylines to match the geologic interpretation.

In some instances, overlaps occur between the generated wireframes for different rock types. These overlaps were addressed by considering the relative age of the geologic units. The assumption was made that, in areas of overlap, the younger unit replaced the older unit. For example, SKN_EN was treated as the younger unit when overlapping the SKN_EX unit. This approach ensures a geologically reasonable interpretation of the model.

Finally, the completed geologic model was compared to the original 20-foot drillhole composites (Table 14-2). This comparison serves as a validation step to verify the model's accuracy. By comparing the model to the actual drillhole data (hard data locations), we can ensure that the implicit modeling method adequately captured the distribution of rock types observed in the field. This validation step helps confirm the model's effectiveness in representing the geological information.

Table 14-2: Geologic Model Comparison to 20-foot Composites

Model Lithology	Model Length	Matching Length	Matching Percent	Not Matching Length	Not Matching Percent
GP	38,647.98	32,391.97	83.8%	6,256.01	16.2%
CRB	17,025.27	14,351.54	84.3%	2,673.73	15.7%
SKN_EX	14,594.60	12,802.46	87.7%	1,792.14	12.3%
GRA	12,779.88	10,407.75	81.4%	2,372.13	18.6%
SKN_MT	10,938.86	10,459.93	95.6%	478.93	4.4%
SKN_EN	9,805.42	9,168.97	93.5%	636.45	6.5%
OVB	5,110.25	2,214.00	43.3%	2,896.25	56.7%
DYK	1,712.99	1,656.12	96.7%	56.87	3.3%
GRA_MT	1,579.09	1,216.30	77.0%	362.79	23.0%
Outside Model	620.27	0.00	0.0%	620.27	100.0%

14.2.2 Estimation Domains

Visual evaluation of assay data in cross-sections revealed that significant mineralization extends beyond the skarn units (magnetite, iron oxide breccia, pyroxene, and garnet). Higher-grade zones were found along sub-parallel structures within these skarns and neighboring lithologies.

To evaluate this mineralization distribution, indicator shells were employed. A copper shell with a cut-off grade of +0.2% Cu was used to define areas of continuous higher-grade copper mineralization. Additionally, indicator shells were created for gold (Au) and silver (Ag) using cut-off values of 0.003 opt and 0.1 opt, respectively. These cut-off grades were chosen to differentiate zones of continuous mineralization from the country rock, with the goal of reducing the coefficient of variation within the resulting domains. To reduce the impact of short-scale data fluctuations, 20-foot length-weighted composites were used to smooth the indicator shells.

The indicator shells were designed to reflect the geological characteristics of mineralization. The search orientation was guided by geological interpretations of lithology, alteration, and structure within the structural domains (Red Star, Empire, AP). To ensure a balanced representation, attention was paid to include a comparable number of data points below the cut-off within the shell as those excluded above the cut-off. This approach aimed to minimize the influence of lower grades within the shell and higher grades

outside, while still allowing for sufficient overlap to represent the gradational nature of the mineralization. Shapes with limited data or small volumes were also removed to ensure a more robust estimation domain.

This process resulted in a set of indicator shells that effectively capture the spatial distribution of the mineralization at Empire, focusing on zones exceeding the pre-defined thresholds for copper, gold, and silver.

14.3 Depletion

To exclude mined-out areas, historical level maps (Farwell & Full, 1944) were used to define stope and mine working extents. Additionally, 3D solids were constructed for intervals logged as voids or mine working. These combined solids were then used to calculate the percentage of mined-out material within each block.

14.4 Compositing

In preparation for resource estimation, 10-foot downhole composites were created from the drillhole database. These composites were then utilized for grade capping analysis and variography to examine data distribution and spatial continuity of mineralization within each domain solid (refer to Table 14-3 for the composite descriptive statistics).

Table 14-3: Domain Composite Data

Metal	Area	Domain	Count	Minimum	Maximum	Mean	Std. Dev.	CV
Cu (%)	AP	Inside	2097	0.002	13.540	0.544	0.736	1.354
		County	5008	0.001	3.705	0.068	0.146	2.153
	Empire	Inside	143	0.002	5.547	0.713	0.921	1.292
		County	2462	0.001	3.803	0.057	0.199	3.485
	Red Star	Inside	77	0.001	2.776	0.466	0.646	1.386
		County	732	0.001	0.907	0.035	0.079	2.286
Ag (g/t)	AP	Inside	2706	0.354	0.001	20.059	0.766	2.166
		County	3180	0.048	0.001	3.938	0.100	2.090
	Empire	Inside	139	0.381	0.009	2.313	0.416	1.090
		County	2441	0.056	0.001	2.421	0.139	2.502
	Red Star	Inside	112	0.908	0.007	15.327	2.091	2.302
		County	697	0.043	0.001	1.440	0.095	2.231
Au (g/t)	AP	Inside	1388	0.0001	0.6505	0.0144	0.0295	2.0482
		County	2206	0.0001	0.0994	0.0014	0.0040	2.8296
	Empire	Inside	138	0.0001	0.2312	0.0121	0.0239	1.9701
		County	2442	0.0001	0.2053	0.0010	0.0052	5.4238
	Red Star	Inside	119	0.0001	0.1128	0.0129	0.0224	1.7457
		County	690	0.0001	0.1123	0.0010	0.0051	4.9854

14.5 Capping

Grade capping is the practice of replacing any statistical outliers with a maximum value from the assumed sampled distribution. This is done statistically to better understand the true mean of the sample population. The estimation of highly skewed grade distribution can be sensitive to the presence of even a few extreme values. HRC utilized a log scale Cumulative Frequency Plot ("CFP") of the composited assay data for each metal to identify the presence of statistical outliers. Capping for each element within the estimation domains was determined from these plots.

A high-grade search distance constraint was also implemented in interpolation. This methodology limits the samples that will be considered to those within a specified distance percentage of the search ellipsoid size, and only those outside that distance if they are within the threshold value. If a sample point is beyond the distance threshold and the point's value exceeds the threshold, it is set to the threshold value. Table 14-4 summarizes the capping strategy used in the estimation process. Distance percentages are based on the total search volume for the domain as defined in Table 14-8.

Table 14-4: Capping Strategy

Area	Domain	Outlier Method	Cu (%)	Ag (g/t)	Au (g/t)
AP	Inside	Cap	4.00	5.00	0.20
		Threshold	1.75	2.00	0.12
		Distance (%)	50%	50%	50%
	County	Cap	1.00	0.60	0.03
		Threshold	---	---	---
		Distance (%)	---	---	---
Empire	Inside	Cap	2.50	1.20	0.10
		Threshold	1.60	1.00	0.05
		Distance (%)	50%	50%	50%
	County	Cap	0.60	0.60	0.02
		Threshold	---	---	---
		Distance (%)	---	---	---
Red Star	Inside	Cap	1.50	6.00	0.08
		Threshold	0.70	---	0.03
		Distance (%)	50%	---	50%
	County	Cap	0.30	0.80	0.02
		Threshold	---	---	---
		Distance (%)	---	---	---

14.6 Variography

A variogram analysis was conducted to quantify the spatial variability of the estimated metals. The resulting variogram model, based on the analysis of distance-based semi variance between samples, serves as an input for ordinary kriging algorithm. This model influences the weights assigned to surrounding data points during block value estimation. Samples closer in proximity and exhibiting higher spatial correlation contribute more significantly to the estimated value of a specific block within the model.

Leapfrog Edge software was used for variogram analysis. The parameters are presented in Table 14-5 and Table 14-6.

Table 14-5: Indicator Domain Variogram Parameters

Domain	Model	Metal	Nugget	C1	C2	R1 (X,Y,Z)			R2 (X,Y,Z)		
AP	Spherical	Cu	0.22	0.33	0.58	50	15	18	125	50	55
	Spherical	Ag	0.20	0.30	0.19	45	65	38	175	175	75
	Exponential	Au	0.33	0.25	0.42	30	28	44	200	200	125
Empire	Spherical	Cu	0.32	0.42	0.26	15	38	25	100	50	70
	Spherical	Ag	0.15	0.27	0.58	50	50	25	165	165	50
	Spherical	Au	0.38	0.36	0.26	30	30	25	250	200	75
Red Star	Spherical	Cu	0.34	0.30	0.36	50	20	20	85	50	63
	Exponential	Ag	0.58	0.42	---	70	45	25	---	---	---
	Spherical	Au	0.49	0.51	---	150	60	30	---	---	---

Table 14-6: Outside and OVB Domain Variogram Parameters

Domain	Metal	Model	Nugget	Sill	R1 (X,Y,Z)		
AP	Cu	Spheroidal	0.44	0.56	200	---	---
	Ag	Spheroidal	0.30	0.70	150	---	---
	Au	Spheroidal	0.44	0.56	200	200	80
Empire	Cu	Spheroidal	0.37	0.63	225	---	---
	Ag	Spherical	0.38	0.62	150	---	---
	Au	Spheroidal	0.20	0.60	100	---	---
Red Star	Cu	Spheroidal	0.42	0.58	25	25	25
	Ag	Spheroidal	0.58	0.42	70	45	25
	Au	Spheroidal	0.58	0.42	85	35	35
OVB	Cu	Spheroidal	0.30	0.70	100	---	---
	Ag	Spheroidal	0.30	0.70	100	---	---
	Au	Spheroidal	0.30	0.70	100	---	---

14.7 Estimation Methodology

The copper, gold and silver grades were estimated from 10-foot down-hole composites using Ordinary Kriging (OK) or a Radial Basis Function (RBF) depending on the domain (Table 14-7). Composites were coded according to the estimation domain.

OK is a geostatistical technique that estimates the value of a property (e.g., grade) at an unsampled location using a weighted average of surrounding samples. The weights are assigned based on the spatial variability of the property, quantified by a variogram model.

RBF is a broader mathematical approach used for interpolation. It produces a function that models the known data and can provide an estimate for any unknown point. In resource estimation, it utilizes spheroidal variogram information to model the spatial variability of the property and then interpolates values at unsampled locations.

Table 14-7: Estimation Algorithm and Search Method

Domain		Metal	Algorithm	Search Method
AP	Inside	Cu	OK	Variable
		Ag	OK	Variable
		Au	OK	Single
	Outside	Cu	RBF	Variable
		Ag	RBF	Variable
		Au	RBF	Single
Empire	Inside	Cu	OK	Variable
		Ag	OK	Variable
		Au	OK	Single
	Outside	Cu	RBF	Variable
		Ag	RBF	Variable
		Au	RBF	Single
Red Star	Inside	Cu	OK	Variable
		Ag	OK	Single
		Au	OK	Variable
	Outside	Cu	RBF	Variable
		Ag	RBF	Single
		Au	RBF	Variable
OVB		Cu	RBF	Variable
		Ag	RBF	Variable
		Au	RBF	Variable

The search volumes were established based on the variograms and the practitioner's experience with similar style projects. The estimation was completed using 3 passes with the same strategy for all Indicator domains (Table 14-8). To account for the variability in mineralization direction within domains due to undulating and branching skarn bodies, a variable orientation approach was adopted for sample search criteria for certain domains. This method allows localized realignment of the search ellipse and variogram based on local geological features, ultimately improving the accuracy of local resource estimates.

Table 14-8: Indicator Domain Estimation Parameters

Copper Indicator Domain Estimation Parameters									
Domain	**AP**			**Empire**			**Red Star**		
# of composites	1st Pass	2nd Pass	3rd Pass	1st Pass	2nd Pass	3rd Pass	1st Pass	2nd Pass	3rd Pass
Min	9	6	3	9	6	3	9	6	3
Max	15	12	9	15	12	9	15	12	9
Max per Hole	3	3	3	3	3	3	3	3	3
Search Ellipsoid Distance									
Primary	110.0	165.0	247.5	80.0	120.0	180.0	80.0	120.0	180.0
Secondary	83.3	125.0	187.4	66.8	100.1	150.2	56.8	85.1	127.7
Tertiary	33.3	50.0	75.0	33.3	50.0	75.0	33.3	50.0	75.0
Silver Indicator Domain Estimation Parameters									
Domain	**AP**			**Empire**			**Red Star**		
# of composites	1st Pass	2nd Pass	3rd Pass	1st Pass	2nd Pass	3rd Pass	1st Pass	2nd Pass	3rd Pass
Min	9	6	3	9	6	3	9	6	3
Max	15	12	9	15	12	9	15	12	9
Max per Hole	3	3	3	3	3	3	3	3	3
Search Ellipsoid Distance									
Primary	116.7	175.1	262.6	110.0	165.0	247.5	46.7	70.0	105.0
Secondary	116.7	175.1	262.6	110.0	165.0	247.5	30.0	45.0	67.5
Tertiary	50.0	75.0	112.5	33.3	50.0	75.0	16.7	25.0	37.5
Domain Indicator Domain Estimation Parameters									
Domain	**AP**			**Empire**			**Red Star**		
# of composites	1st Pass	2nd Pass	3rd Pass	1st Pass	2nd Pass	3rd Pass	1st Pass	2nd Pass	3rd Pass
Min	9	6	3	9	6	3	9	6	3
Max	15	12	9	15	12	9	15	12	9
Max per Hole	3	3	3	3	3	3	3	3	3
Search Ellipsoid Distance									
Primary	133.3	200.0	300.0	166.8	250.1	375.2	100.0	150.0	225.0
Secondary	133.3	200.0	300.0	133.3	200.0	300.0	40.0	60.0	90.0
Tertiary	83.3	125.0	187.5	50.0	75.0	112.5	20.0	30.0	45.0

14.8 Density

The following discussion of the density specific to the Project is largely modified from, and in some cases, is excerpted directly from the 2017 SRK report.

Density measurements of unaltered material were applied from literature research (Berkman, 1989). Oxidized densities were derived from a combination of data from metallurgical reports of in-pit bulk samples completed by Kappes, Cassidy & Associates (“KCA”) in 2013 and from a 2017 campaign of density determinations directed by SRK and carried out by Konnex (n = 83). Konnex used ASTM C914 – Standard Test Method for Rock Density and Volume of Solid Refractories by Wax Immersion. This method was adopted by Konnex from KCA for consistency.

The resultant density database consists of 99 measurements, with an average SG of 2.95. A total of 18-20 samples were averaged for each of the mineralized rock types (Table 14-9). There was a strong correlation between 2017 density determination by Konnex and densities from KCA.

Table 14-9: Modeled Density Factors

	Rock Code	ton/ft^3	ft^3/ton
Qal	10	0.062	16.18
QFP	12	0.081	12.32
FeOxBx	20	0.076	13.13
Garnet Skarn	30	0.111	9.00
Pyroxene Skarn	32	0.102	9.80
Mt Skarn	34	0.144	6.96
Limestone	51	0.084	11.91
Granite	60	0.078	12.81
Dike	61	0.078	12.87

As stated in the previous reports, the oxidation state is a critical component to modeling because it affects both the acid-leach recovery and the density of the material. The density has been shown to vary with the degree of oxidation, but the oxidation state is not uniformly addressed in the geologic data collected for the Project. HRC utilized an average density of the oxidized material for the updated resource estimate. This is a conservative approach to be used until such time as the degree of the oxidation can be quantified.

14.9 Validation

HRC utilized several methods to validate the results of the estimation method. The combined evidence from these validation methods verifies the OK estimation model results.

14.9.1 Comparison with Nearest Neighbor Models

Nearest Neighbor (NN) and Inverse Distance (ID) models were run to serve as comparison with the estimated results from the OK method. Descriptive statistics for the OK method along with those for the NN, ID, and drill hole composites for the indicator domains are shown in Table 14-10.

Table 14-10: Model Comparison Descriptive Statistics

Copper (%)						
Estimate	Count	Minimum	Maximum	Mean	Std. Dev.	CV
Composites	2,317	0.001	13.540	0.552	0.747	1.354
Nearest Neighbor	17,147	0.001	13.540	0.543	0.351	0.647
Ordinary Krige	17,281	0.026	2.741	0.543	0.351	0.647
Inverse Distance	17,037	0.012	3.747	0.543	0.351	0.647
Silver (opt)						
Estimate	Count	Minimum	Maximum	Mean	Std. Dev.	CV
Composites	2,957	0.001	20.059	0.376	0.849	2.258
Nearest Neighbor	39,960	0.001	20.059	0.349	0.685	1.960
Ordinary Krige	39,412	0.025	3.555	0.332	0.270	0.814
Inverse Distance	39,412	0.012	4.891	0.334	0.340	1.019
Gold (opt)						
Estimate	Count	Minimum	Maximum	Mean	Std. Dev.	CV
Composites	1,645	0.0001	0.6505	0.0141	0.0286	2.0289
Nearest Neighbor	16,603	0.0001	0.6505	0.0143	0.0236	1.6479
Ordinary Krige	16,513	0.0009	0.0733	0.0137	0.0082	0.6026
Inverse Distance	16,513	0.0002	0.1680	0.0140	0.0104	0.7407

The overall reduction of the maximum and standard deviation within the OK model represents an appropriate amount of smoothing to account for the point to block volume variance relationship while maintaining similar means.

14.9.2 Swath Plots

Swath plots were generated to compare average estimated grade from the OK and ID methods to the NN and validation model. The results from the OK model are compared using the swath plot to the distribution derived from the NN model.

Three swath plots were generated for each element. Swath plots for copper are presented as an example of the results: Figure 14-3 shows average copper grade from west to east; Figure 14-4 shows average copper grade from south to north, and Figure 14-5 shows average copper grade from bottom to top.

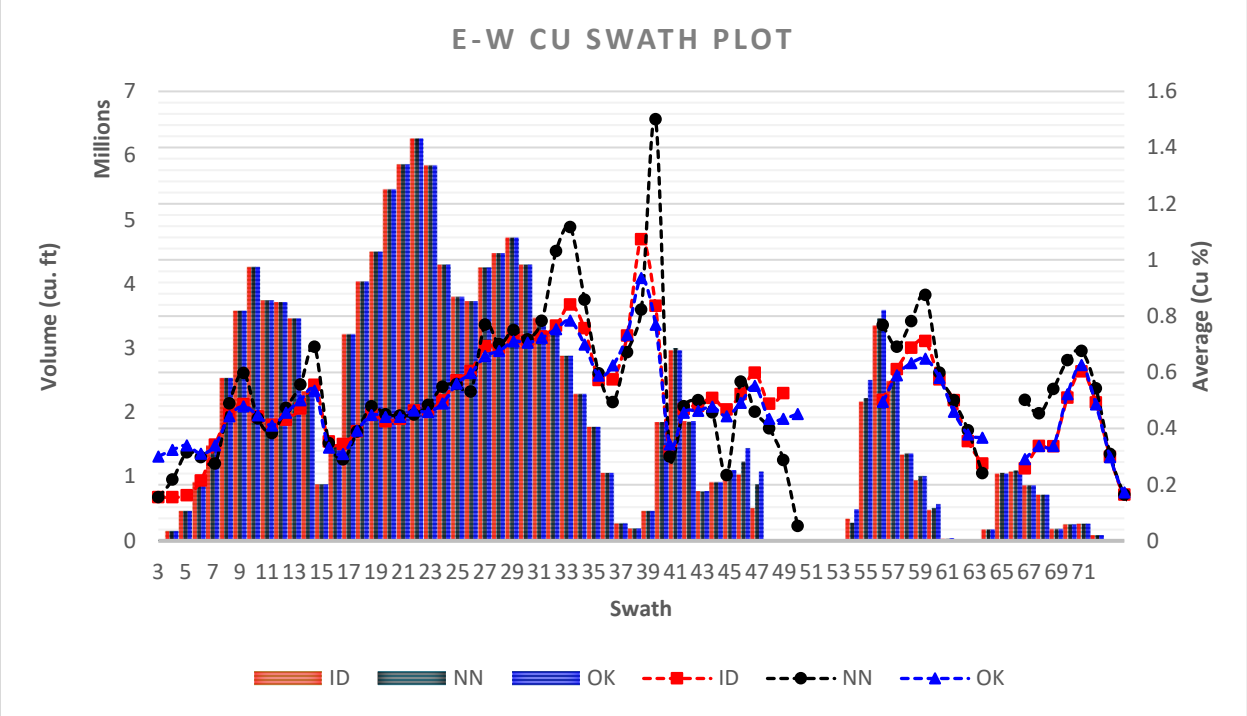


Figure 14-3: East-West Copper Swath Plot

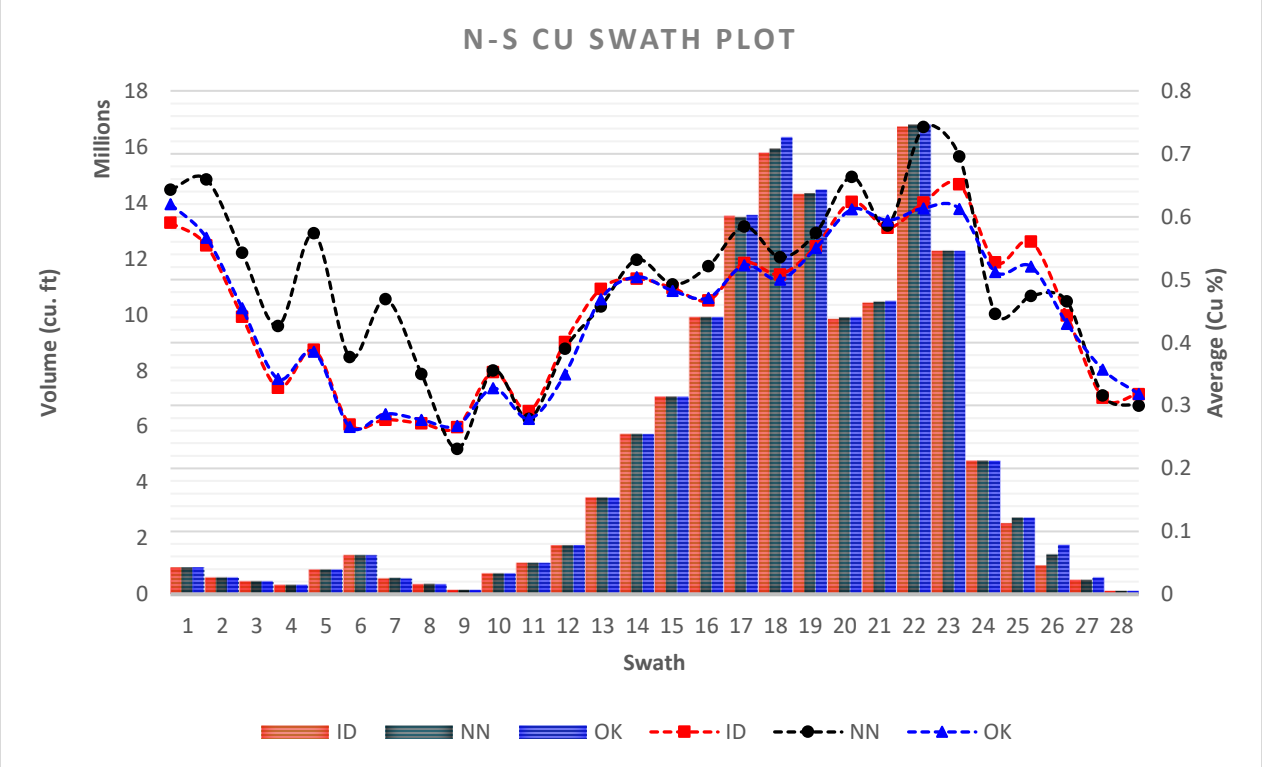


Figure 14-4: North-South Copper Swath Plot

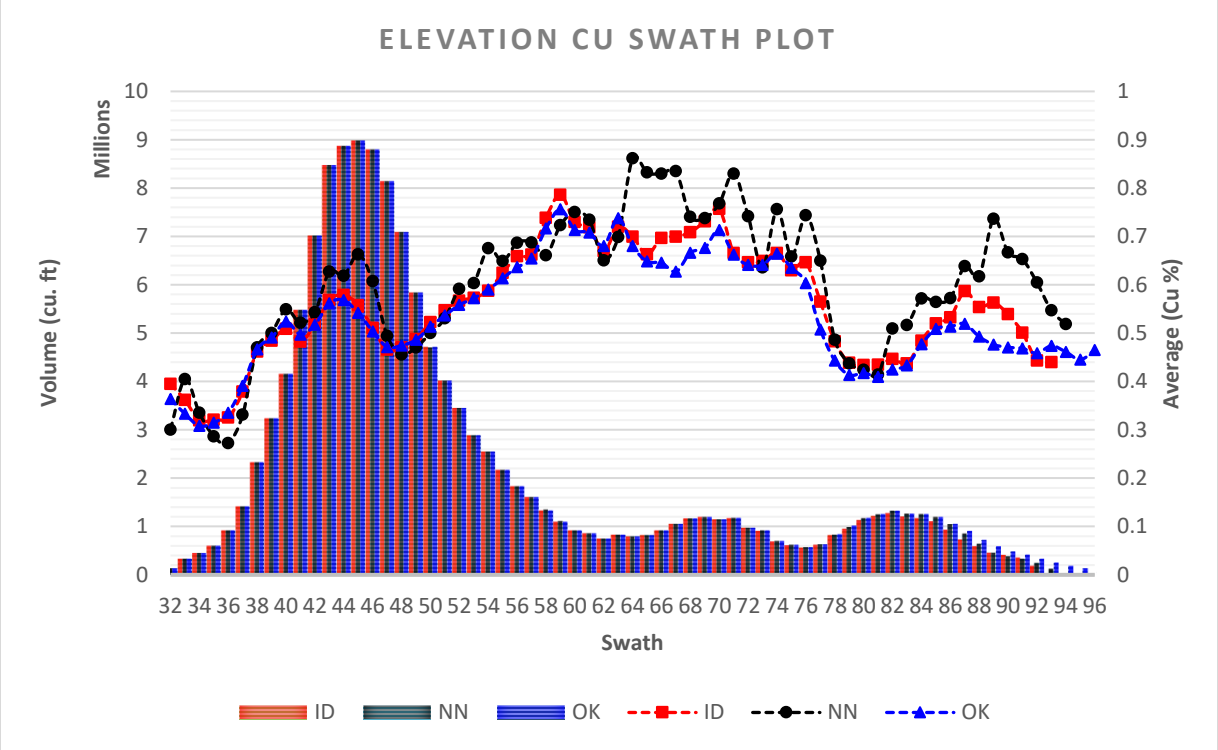


Figure 14-5: Elevation Copper Swath Plot

14.9.3 Section Inspection

Bench plans, cross-sections, and long sections comparing modeled grades to the 20-foot composites were evaluated. The example sections displaying estimated copper grades are shown in Figure 14-7 through Figure 14-9. The figures show good agreement between modeled grades and the composite grades. In addition, the modeled blocks display continuity of grades along strike and down dip.

On a local scale, the nearest neighbor model does not provide a reliable estimate of grade, but on a much larger scale, it represents an unbiased estimation of the grade distribution based on the total data set. Therefore, if the OK model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the distribution of grade from the nearest neighbor.

Correlation between the grade models is generally good, though deviations occur near the edges of the deposit and in areas where the density of drilling is lesser, and material is classified as Inferred resource.

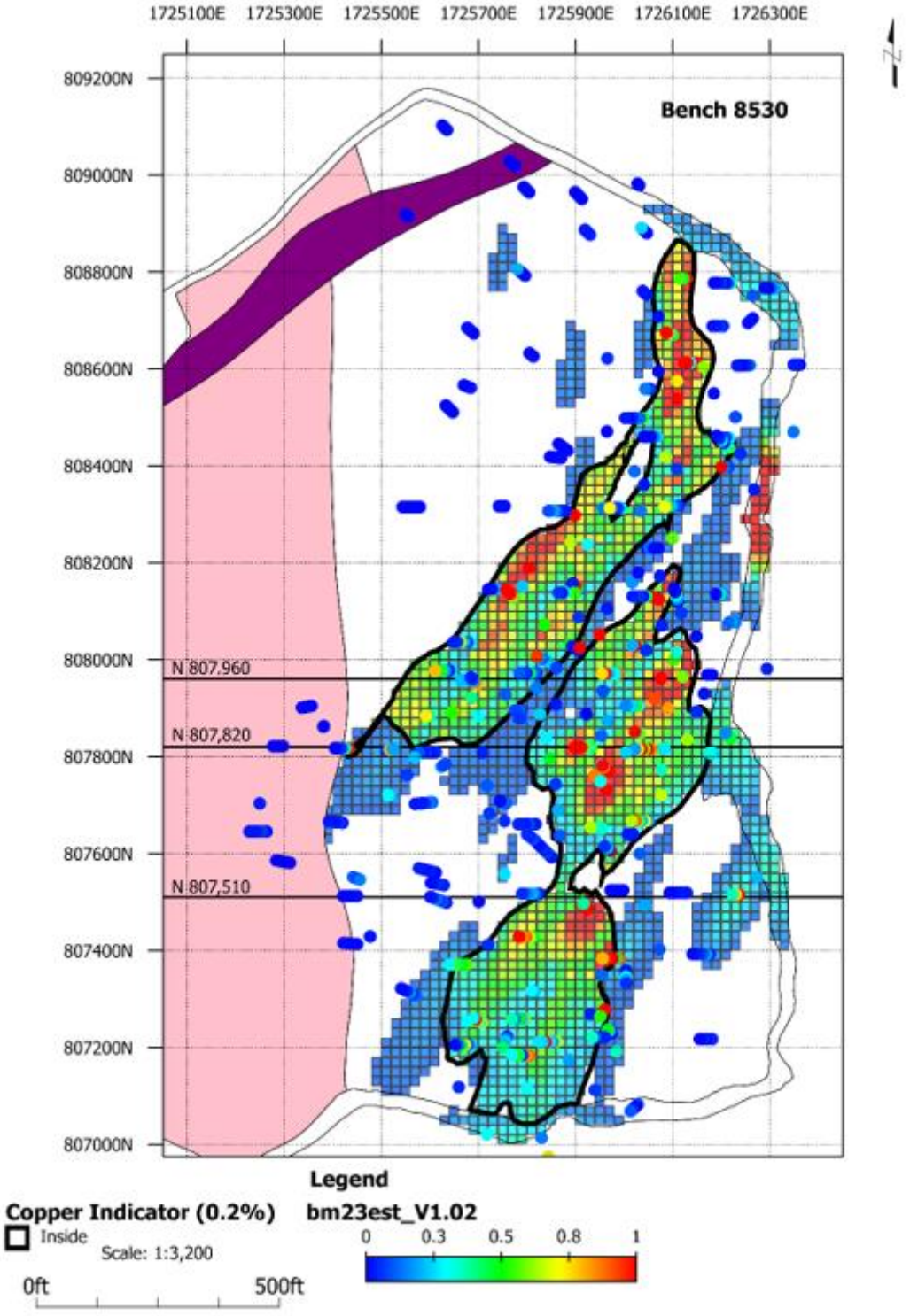


Figure 14-6: Bench 8530 of Estimated Copper Grades with Composites and Section Location Lines

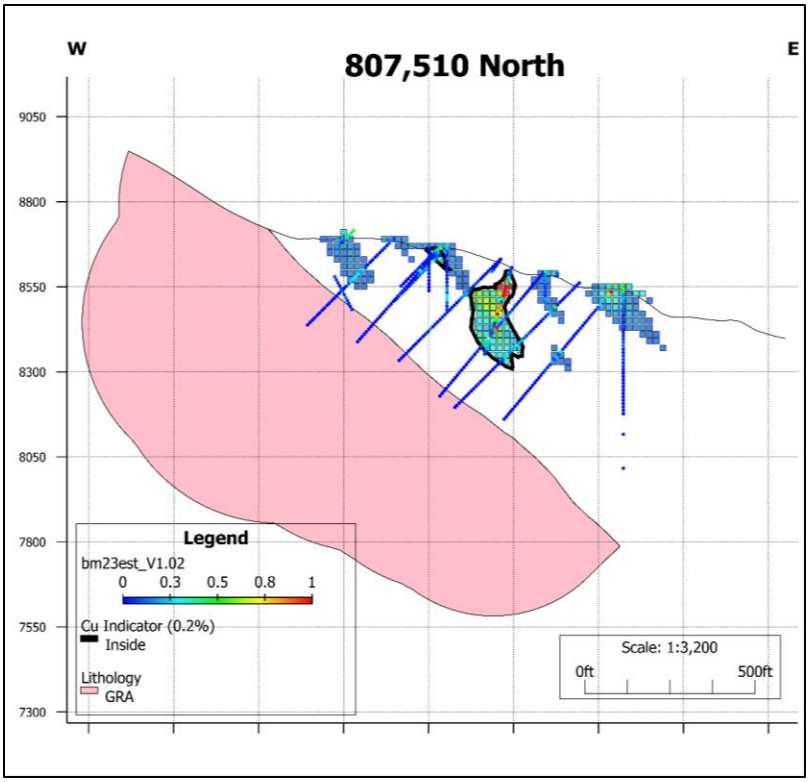


Figure 14-7: N807,510 Cross Section of Estimated Copper Grades with Composites

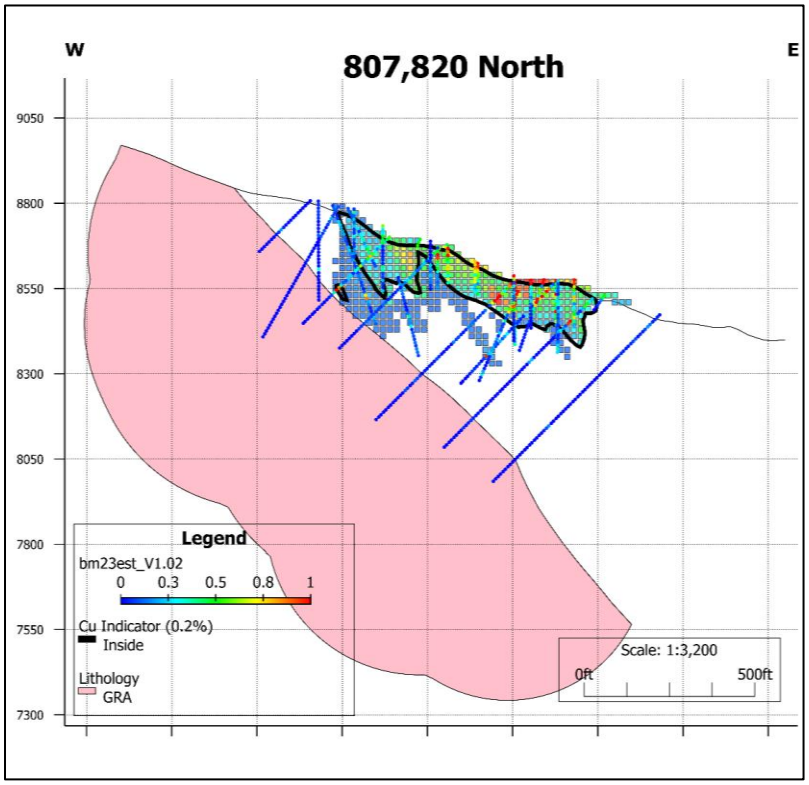


Figure 14-8: N807,820 Cross Section of Estimated Copper Grades with Composites

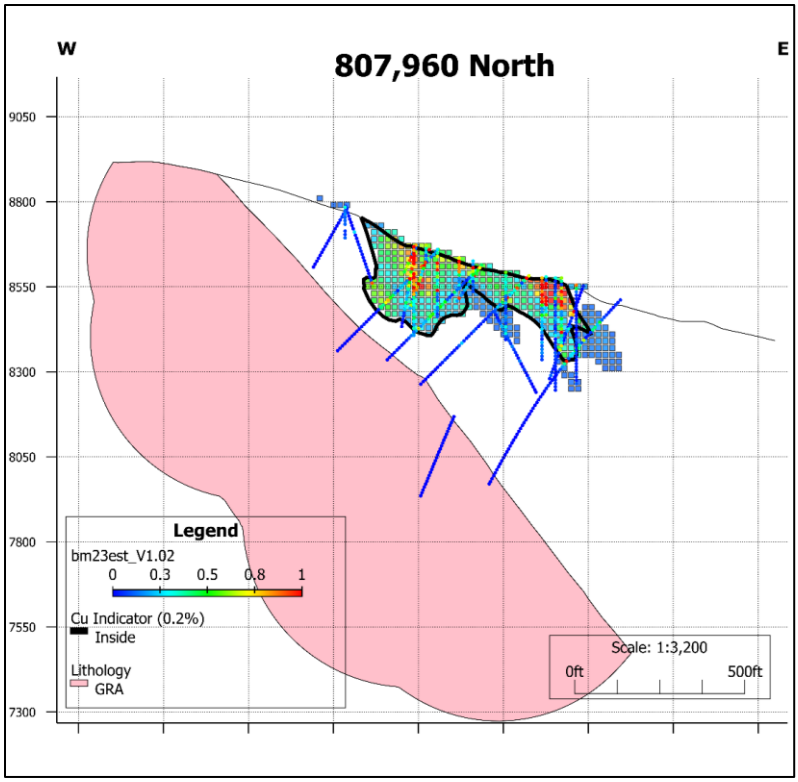


Figure 14-9: N807,960 Cross Section of Estimated Copper Grades with Composites

14.10 Mineral Resource Classification

HRC classified mineral resources as Measured, Indicated, or inferred based on two factors: domain code and estimation parameters. Domain code identifies the geological classification (e.g., indicator zone, alluvium). Blocks coded as alluvium or outside of the indicator domains are automatically classified as Inferred due to their uncertain resource potential.

Estimation parameters, detailed in Table 14-9, define the criteria used for resource estimation Measured resources were classified in two steps depending on the estimation metal. First blocks that reside within both indicator domains for Cu and Ag and were estimated in the first pass for both metals were classified as measured. Secondly, blocks located within the Au indicator domain and estimated in the first pass were added to the measured classification.

The remaining blocks estimated in the 1st pass or 2nd passes with in any indicator domain were classified as indicated. Finally, all blocks estimated in the 3rd pass were classified as inferred.

14.10.1 Mineral Resource Tabulation

In order to identify the mineralization that meets the test for reasonable prospects of eventual economic extraction, and thus be classified as mineral resources, a Lerchs-Grossman pit shell was generated. The optimization parameters utilized the net smelter return (NSR) values calculated in each block. The NSR calculation takes into account the estimated metal recovery for each metal, the treatment charges, payment terms, deducts, and shipping charges. The operating costs were determined based on HRC's and Forte's industry knowledge and prior experience. Pit slope angles were restricted to 44 to 50 degrees. The optimization parameters are presented below in Table 14-11. Blocks classified as Measured, Indicated, and Inferred were used to define the resource pit shell.

Table 14-11: Parameters used for Resource Pit Shell Generation

Resource Parameter	Units	Cu	Au	Ag
Commodity Prices	**$/oz or $/lb**	**$4.25**	**$1,900**	**$25.50**
Flotation Process Recoveries				
Flotation _ Cu Concentrate	%	33.0%	50.0%	36.0%
Concentrate (Payables)				
Flotation_ Cu Concentrate (Au Payable based on grade)	%	95.0%	90-97%	95.0%
Cementation Process Recoveries				
Cementation (Total Copper Recovery after Flotation)	%	90.0%	0.0%	0.0%
Cementation Payable	%	100.0%	0.0%	0.0%
Treatment/Refining Charges				
Copper Con. Refining	Ag $/oz	0.40		
Copper Con. Refining	Au $/oz	4.00		
Copper Con. Trucking & Shipping $/t conc	wet	$80.00		
Copper Con. Treatment $/t conc	wet	$90.00		
Copper Cementation Shipping $/lb	Cu $/lb	$0.04		
Copper Cementation Selling $/lb	Cu $/lb	$0.02		
Operating Costs				
Mining Cost – Surface	$/t mined	$2.32		
Mining Cost - Incremental Increase for each 20ft depth	$/t mined	$0.016		
Processing Cost	$/t milled	$17.00		
G&A	$/t milled	$2.00		
Total Ore cost $/t milled	$/t milled	$19.00		
Pit Slope Assumptions	Five sectors were modelled based on core logging with inter-ramp angles ranging from 44º to 50º			

The cutoff used to report resources inside the optimized pit shell is based on a $19.00/t NSR cutoff. The estimated plant operating costs were provided by Forte and the cutoff reflects the potential economic, marketing, and other issues relevant to an open pit mining scenario based on a milling recovery process producing a copper concentrate and cemented copper. Note that the mining costs are not included in the cutoff calculation as an internal cutoff is used and the mining costs are considered a sunk cost.

Table 14-12: Resource Cutoff Parameters

Department		Cost
Processing	$/ore ton	$ 17.00
Production Taxes	$/ore ton	$ 0.25
G&A	$/ore ton	$ 1.75
Total Costs (NSR Cutoff)	**$/ore ton**	**$ 19.00**

14.11 Mineral Resource Statement

The mineral resource estimate for the Empire Project is summarized in Table 14-13. This mineral resource estimate includes all drill data obtained as of January 1, 2024, and has been independently verified by HRC. Mineral resources are not mineral reserves and may be materially affected by environmental, permitting, legal, socio-economic, marketing, political, or other factors. The measured and indicated mineral resources are inclusive of the mineral reserves. Inferred mineral resources are, by definition, always additional to mineral reserves.

Table 14-13: Mineral Resource Statement for the Empire Mine, Custer County, Idaho, U.S.A., Hard Rock Consulting, LLC, January 1, 2024

Classification	Tons	Copper		Gold		Silver		Copper Equiv.	
	(x1000)	%	lb (x1000)	opt	oz (x1000)	opt	oz (x1000)	%	lb (x1000)
Measured	8,467	0.48	80,643.61	0.012	98.9	0.41	3,497.4	0.67	113,590.89
Indicated	3,120	0.50	30,989.95	0.005	15.2	0.41	1,272.1	0.61	37,824.15
Measured + Indicated	**11,587**	**0.48**	**111,633.57**	**0.010**	**114.1**	**0.41**	**4,769.5**	**0.65**	**151,415.04**
Inferred	355	0.57	4,043.13	0.005	1.9	0.17	59.2	0.66	4,666.06

Notes:

(1) The effective date of the 2024 Mineral Resource estimate is January 1st, 2024. The QP for the estimate is Zachary J. Black, RM-SME, of Hard Rock Consulting, LLC.

(2) Mineral resources are not mineral reserves and do not have demonstrated economic viability.

(3) Mineral Resources are reported inclusive of Mineral Reserves at $19.00 NSR cut-off. The Mineral Resources are considered amenable to open pit mining and are reported within an optimized pit shell. The pit optimization is based on total ore costs of $19.00/t processed, mining costs of $2.32/t at surface and increasing $0.016/t for every 20ft of depth and pit slope angles ranging from 44 to 50 degrees.

(4) Cu-Eq (copper equivalent grade) is based on the mill recoveries and metal prices shown in Table 14-11.

(5) Copper Equivalent (CuEq) = ((Cu head grade x recovery x Cu Price) + (Au head grade x recovery x Au Price) + (Ag head grade x recovery x Ag Price)) / (Cu recovery x Cu Price)

(6) Tonnage and grade estimates are in Imperial units.

(7) Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

14.11.1 Resource Sensitivity

The block model tons and grades are shown in Figure 14-10 at variable copper equivalent prices within corresponding pits.

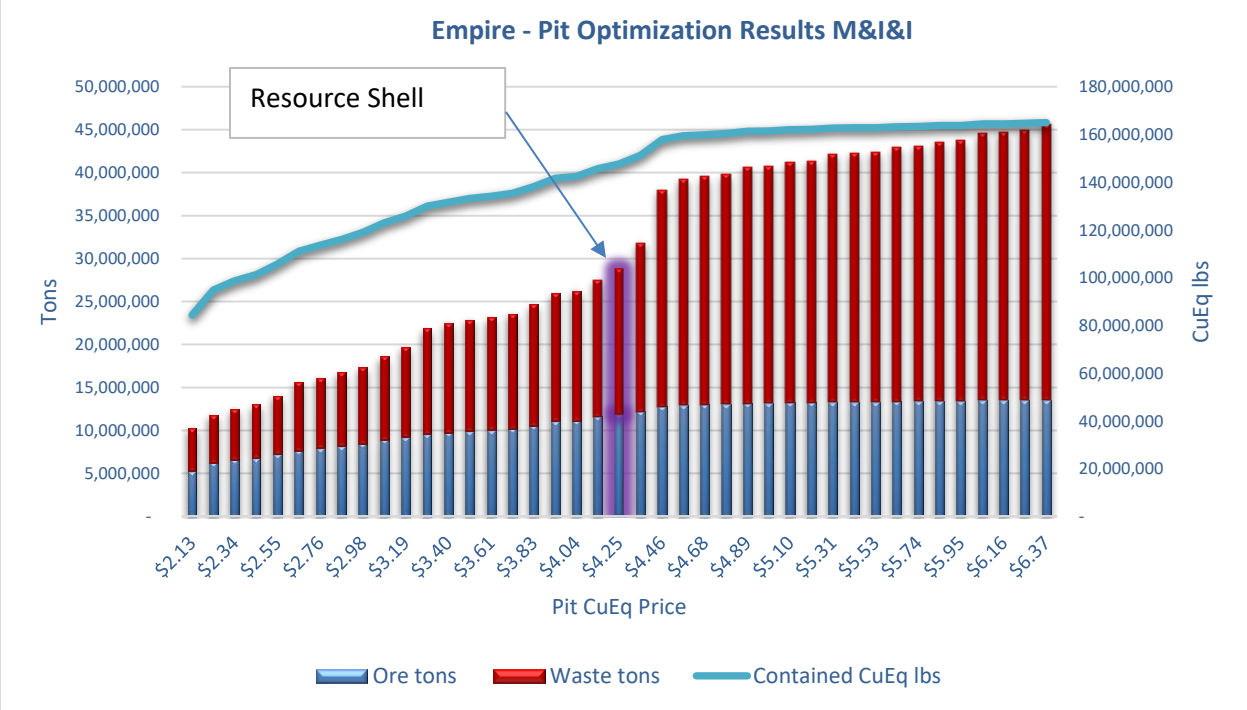


Figure 14-10: Pit Optimization Copper Sensitivity Chart

15. MINERAL RESERVES

15.1 Introduction

Mineral Reserves were estimated for the Empire project which is planned to be an open pit operation with a flotation mill and a cementation plant for processing. All estimates are based on the mine plans generated by HRC for the pre-feasibility study which include three phases for the open pit.

15.2 Estimation Parameters

Open pit mine designs have been completed on the Empire deposit which have been evaluated with the updated Measured and Indicated Resources and have been demonstrated to be economically viable, therefore Measured and Indicated Mineral Resources within the pit designs have been converted to Proven and Probable Reserves. The Mineral Reserves use the terminology, definitions and guidelines given in the CIM Standards on Mineral Resource and Mineral Reserves (May 2014). All inferred material has been classified as waste and scheduled to the appropriate waste dump. The final open pit reserves are reported using a $20.50 NSR cutoff inside the pit design which is discussed in more detail the following sub-sections along with the pit deigns parameters.

15.2.1 Pit Slopes

The pit slopes follow recommendations provided by Call & Nicholas Inc. (CNI). CNI provided a provided a pre-feasibility level pit slope study in 2017 and an update to the study in 2022 wherein data and analysis gaps identified as part of CNI's 2017 study were addressed. The design criteria and inter-slope angles used for the pit optimizations and pit designs are presented in Table 15-1. The overall pit slope design criteria utilized for slope stability analysis was a 1.2 static and 1.0 pseudo-static factor of safety with respect to overall slope failure. The 60' triple benches utilized a 27' catch bench and the 40' double benches utilized a 23' catch bench with an 80 percent reliability.

Table 15-1: Recommended Inter-Ramp Slope Angles

Rock Type	Bench Height (ft)	Wall Dip Direction (deg)	ISA (deg)
Intrusives	40	All	44
	60	All	50
FeOxBx	40	All	44
Limestone	40	0-150, 240-360	43
		150-240	39
	60	0-150, 240-360	49
		150-240	47

15.2.2 Dilution and Losses

The mineral resource estimate for Empire is considered to be internally diluted by compositing. HRC also calculated an external diluted grade for all of the grade elements; these diluted grades were used for the mineral reserve calculation. To estimate the external dilution, each side of every ore block was queried to determine if it had a waste block adjacent to each side. If the adjacent block was determined to be waste, then 16.7% of the waste block was included in a weighted average grade estimate for the block. The 16.7% of the waste block is calculated based on a wedge with an 8-foot-wide bottom included as dilution. If two sides of the block are adjacent to waste then the dilution percent is 28.6%, three sides would be 37.5% and all four sides would be 44.4%.

This was applied to all metals and on average, the dilution amounted to 10.3% percentage for the mineral reserve pit design above the $19.00/t NSR. The average grades decreased by approximately 5% and the metal content went up by 4.7% due to the inclusion of the metal content from the dilution material. These percentages were then applied to the $20.50/t NSR cutoff that was used for reporting the Final Mineral Reserves.

15.2.3 Pit Optimization and NSR Calculation

The Mineral Resources for the deposit were evaluated using a Lerchs–Grossmann (L–G) pit optimizer to generate optimized pit shells. Pit shells were generated based on varying metal prices with a base Copper price of $4.00/ lb, a $1788/oz gold price and a $24/oz silver price. A total of 51 pit shells were generated to determine optimal break points for developing pit phases and for determining the ultimate final pit phase for the deposit.

Table 15-2 shows the cost and slope parameters used for each optimization. The operating costs were determined based on HRC's and Forte's industry knowledge and prior experience. In order to apply the cutoff for the tabulation of the mineral reserve, each block in the mineral resource model was assigned an NSR (Net Smelter Return) value calculated in $/ton. As shown in Table 15-2, the same smelter parameters were used for defining the Mineral Reserves as were used for the Mineral Resources

Table 15-2: Reserve Pit Optimization Parameters

Reserve Parameter	Units	Cu	Au	Ag
Commodity Prices	**$/oz or $/lb**	**$4.00**	**$1,788**	**$24.00**
Flotation Process Recoveries				
Flotation _ Cu Concentrate	%	33.0%	50.0%	36.0%
Concentrate (Payables)				
Flotation_ Cu Concentrate (Au Payable based on grade)	%	95.0%	90-97%	95.0%
Cementation Process Recoveries				
Cementation (Total Copper Recovery after Flotation)	%	90.0%	0.0%	0.0%
Cementation Payable	%	100.0%	0.0%	0.0%
Treatment/Refining Charges				
Copper Con. Refining	Ag $/oz	0.40		
Copper Con. Refining	Au $/oz	4.00		
Copper Con. Trucking & Shipping $/t conc	wet	$80.00		
Copper Con. Treatment $/t conc	wet	$90.00		
Copper Cementation Shipping $/lb	Cu $/lb	$0.04		
Copper Cementation Selling $/lb	Cu $/lb	$0.02		
Operating Costs				
Mining Cost – Surface	$/t mined	$2.32		
Mining Cost - Incremental Increase for each 20ft depth	$/t mined	$0.016		
Processing Cost	$/t milled	$17.00		
G&A	$/t milled	$2.00		
Total Ore cost $/t milled	$/t milled	$19.00		
Pit Slope Assumptions	Five sectors were modelled based on core logging with inter-ramp angles ranging from 42º to 45º			

Figure 15-1 shows the optimization results for the Measure and Indicated material within the resource model. Values in the figures are based on optimized pit shells before the design process and do not include the haulage ramps and catch benches.

The Final pit was limited to a $4.00/ lb CuEq pit shell. The pit were designed into three phases for the Mineral Reserve evaluation with the fist phase being based on the $1.52/lb CuEq pit shell and the second phase was based on a $2.00/lb CuEq pit shell.

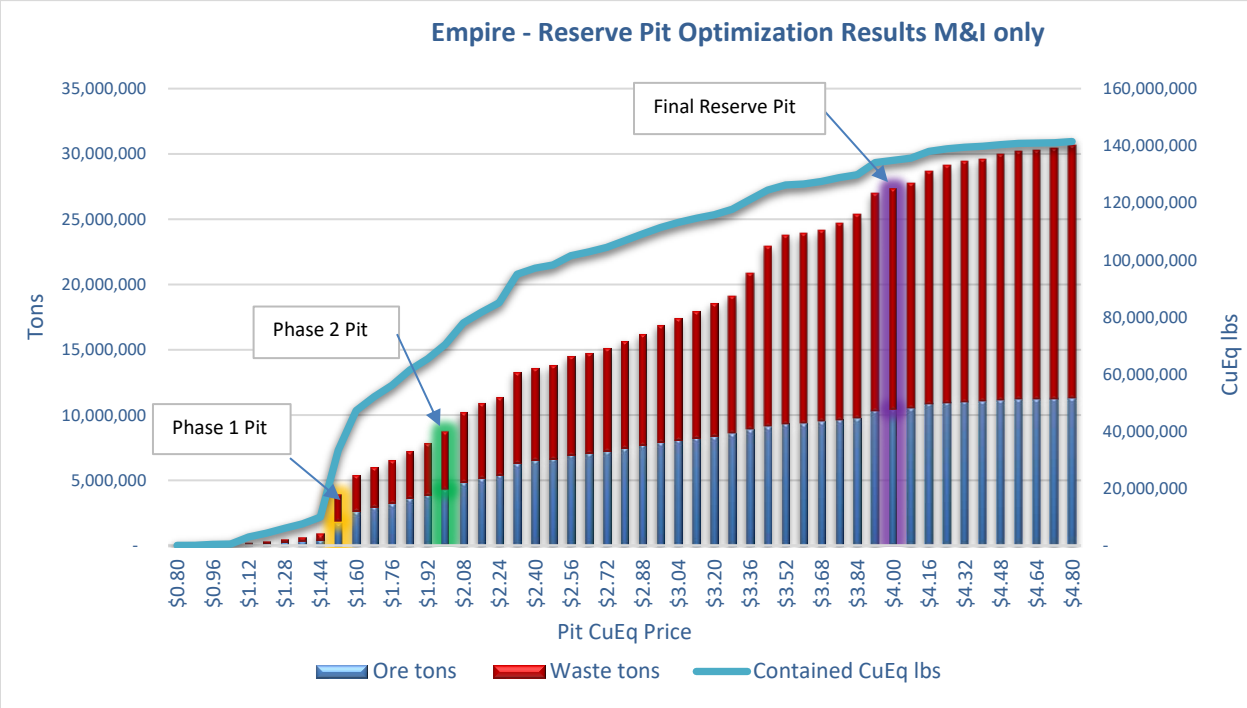


Figure 15-1: Pit Optimization Results

The mineral reserves are reported using a $20.50/t NSR cutoff inside of the final pit design which includes the estimated plant operating costs, all G&A costs, Royalties and the low-grade ore stockpile re-handle costs during pit operation and shown on Table 15-3. The cutoff was evaluated in $0.50/t increments in order to maximize the NPV@7.5% of the Project, the $20.50/t cutoff was found to produce the best results for the NPV calculation.

Table 15-3: Mineral Reserve NSR Cutoff

	NSR Cutoff, $/t
Process Cost (including tailings management)	$15.75
Property G&A Costs and Mining Tax, per ton of ore	$2.25
Royalty Costs, per ton of ore	$1.75
Low Grade Stockpile Rehandle, per ton of ore	$0.75
Total Cost per Ton Ore	**$20.50**

15.3 Mineral Reserve Statement

The Mineral Reserve estimates are reported using the 2014 CIM Definition Standards. The QP for the estimate is Mr. Jeffery Choquette P.E.

The Mineral Reserves are reported using a $20.50/t NSR cutoff inside of the final pit design. Proven and Probable Mineral Reserves of 11.1 million tons are reported within the final pit design shown Table 15-4. All inferred material was classified as waste and will be scheduled to the appropriate waste stockpile. The mineral reserve estimate takes into consideration metallurgical recoveries, concentrate grades, transportation costs, smelter treatment charges and royalties in determining NSR values. Table 15-4 also shows the mineral reserves by classification category and grade.

Table 15-4: Mineral Reserve Statement for the Empire Mine, Custer County, Idaho, U.S.A., Hard Rock Consulting, LLC, January 1, 2024

Diluted Grades	Tons	Copper		Gold		Silver		Copper Equiv.	
Classification	(x1000)	%	lb (x1000)	opt	oz (x1000)	opt	oz (x1000)	%	lb (x1000)
Proven	8,284	0.49	81,070.56	0.011	90.9	0.42	3,483.7	0.68	111,995.19
Probable	2,846	0.50	28,417.41	0.005	13.2	0.41	1,170.7	0.61	34,498.69
Proven + Probable	11,130	0.49	109,487.97	0.009	104.0	0.42	4,654.4	0.66	146,493.88

Notes:

(1) The Mineral Reserves have an effective date of January 1st, 2024. The QP for the estimate is Jefferey Choquette, P.E., of Hard Rock Consulting, LLC.

(2) Mineral reserve tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

(3) All reserves are stated above a $20.50 NSR cutoff and bound within the final pit design.

(4) Tonnage and grade estimates are in Imperial units

(5) Cu-Eq values are based on the metal prices and recoveries presented in Table 15-2.

(6) Copper Equivalent (CuEq) = ((Cu head grade x recovery x Cu Price) + (Au head grade x recovery x Au Price) + (Ag head grade x recovery x Ag Price)) / (Cu recovery x Cu Price)

(7) Rounding as required by reporting guidelines may result in summation differences.

15.4 Factors that May Affect the Mineral Reserve

Areas of uncertainty that may materially impact the Mineral Reserves include the following:

- Variations in the forecast commodity price;
- Variations to the assumptions used in the constraining Lerch Grossman pit shells, including mining loss/dilution, metallurgical recoveries, geotechnical assumptions including pit slope angles, and operating costs;
- Variations in assumptions as to permitting, environmental, and social license to operate.

16. MINING METHODS

16.1 Overview

The Empire mine contains mineralization at or near the surface that is ideal for open pit mining methods. For this PFS, a base case 4,000 tpd flotation mill followed by a cementation plant to leach the oxidized copper has been chosen for the evaluation. The mine plan is based on proven and probable mineral reserves only. The mill flotation circuit will produce a copper concentrate, with contained payables for silver, gold, and copper and the cementation circuit will produce precipitate copper which will only be payable for copper.

A total of 11.1 million tons will be processed during the 8-year mine life. The mine plan is based on proven and probable mineral reserves only. 18.8 million tons will be sent to the waste rock storage facility which results in an overall life of mine (LOM) strip ratio of 1.69:1.

The method of material transport at the Project is expected to be open pit mining using two 7.5-yd3 front end loaders as the main loading units. The open pit mineralized material is loaded into 45-ton articulated haul trucks and transported to the crushing site next to the mill, waste material will be transported to the designated rock waste dump. A maximum of six, 45-ton haul trucks will be required to meet the production schedule over the LOM.

The mine production schedule is based on a seven day per week schedule, with two 12 hour shifts per day. There are four crews planned to cover the rotating schedule. Each 12-hour shift will contain a half hour down for blasting and miscellaneous delays, a half hour for shift start up and shutdown and an hour for lunch and breaks for a total of 10.5 effective working hours. Peak mineralized material and waste production is estimated at 15,350 tpd with an average life of mine rate of 10,000 tpd.

16.2 Geotechnical

In March 2018 Call & Nicolas Inc. (CNI) completed a prefeasibility pit slope study on the Empire Mine Project. The purposes of the study were to recommend design interramp slope angles and bench design parameters for a pre-feasibility level study based on the data available as of December 2017. Stability analyses conducted included: (1) bench-scale analyses from which the expected distribution of bench-face angles (BFA) and reliability schedules were developed, and (2) overall wall stability associated with estimated rock mass strengths. The bench-scale analysis relied on cell-mapping data collected along exposed rock outcrops and historic underground mapping conducted by the USGS in 1944. The overall analyses were performed using mapped fault data from a 1944 USGS report, rock strength data collected in 2017, and drilling data to characterize the rock mass and estimate global rock mass strengths.

In 2022 CNI performed an updated geotechnical pit slope wherein data and analysis gaps identified as part of CNI's 2017 pre-feasibility study were addressed. The following work was done to address technical gaps identified in 2017:

- Geomechanically logged and processed structural televiewer data on two geotechnical drill holes with a total of 1433 feet.
- Collected and tested core from the two geotechnical drill holes.
- Adjusted historic RQD measurements based on statistical analysis to more closely match actual measured conditions.
- Developed RQD block model based on updated historic and new RQD measurements.

- Updated the bench-scale stability analysis with recent televiewer data.
- Ran overall stability analyses on key sections of the proposed pit with additional analyses looking at potential seismic hazard to the slope.

Uniaxial and triaxial laboratory testing added confidence to the material properties used in the overall and bench-scale analyses. Triaxial tests were not originally performed as part of the 2017 prefeasibility study and the 2021 laboratory testing campaign confirmed assumptions used in the previous study.

Overall stability analyses were run on major wall orientations with the updated rock-mass strengths, and pseudo-static conditions were applied based on the maximum considered event (MCE). The results of the overall stability analyses did not largely change from the 2017 study, and the pit is still primarily bench-scale controlled. Bench-scale stability analyses were run with the additional televiewer data collected as part of the 2021 drilling campaign.

The overall pit slope design criteria utilized for updated slope stability analysis was a 1.2 static and 1.0 pseudo-static factor of safety with respect to overall slope failure. The 60' triple benches utilized a 27' catch bench and the 40' double benches utilized a 23' catch bench with an 80 percent reliability. The recommended interramp slope angles used for the design are presented in Table 16-1 and Figure 16-1.

Table 16-1: Recommended Inter-Ramp Slope Angles

Rock Type	Bench Height (ft)	Wall Dip Direction (deg)	ISA (deg)
Intrusives	40	All	44
	60	All	50
FeOxBx	40	All	44
Limestone	40	0-150, 240-360	43
		150-240	39
	60	0-150, 240-360	49
		150-240	47

The geotechnical data collected from the 2018 and 2022 programs is considered sufficient for a PFS level pit slope design however, the following tasks are required should the project advance to the feasibility study and/or detailed engineering stages:

- Only two of six recommended geotechnical drill holes were completed, as the drilling campaign ended early due to inclement weather. The slope angles presented should be regarded as interim recommendations and are based on the available data. The completion of the drilling campaign and a re-evaluation of the pit slopes is recommended.
- Investigate potential differences between the different intrusive units based on the remaining geotechnical drill holes.
- Conduct cell mapping on any exposed bench faces to collect additional orientation, spacing, and length characteristics of structures in the rock fabric.
- Revise slope stability analyses and pit slope recommendations utilizing the improved information.
- Perform magnetite skarn lab testing.
- Further characterize the altered zone found in the southern pit. This zone is also referred to as iron-oxide breccia and is generally low strength in character and is a general geotechnical concern.

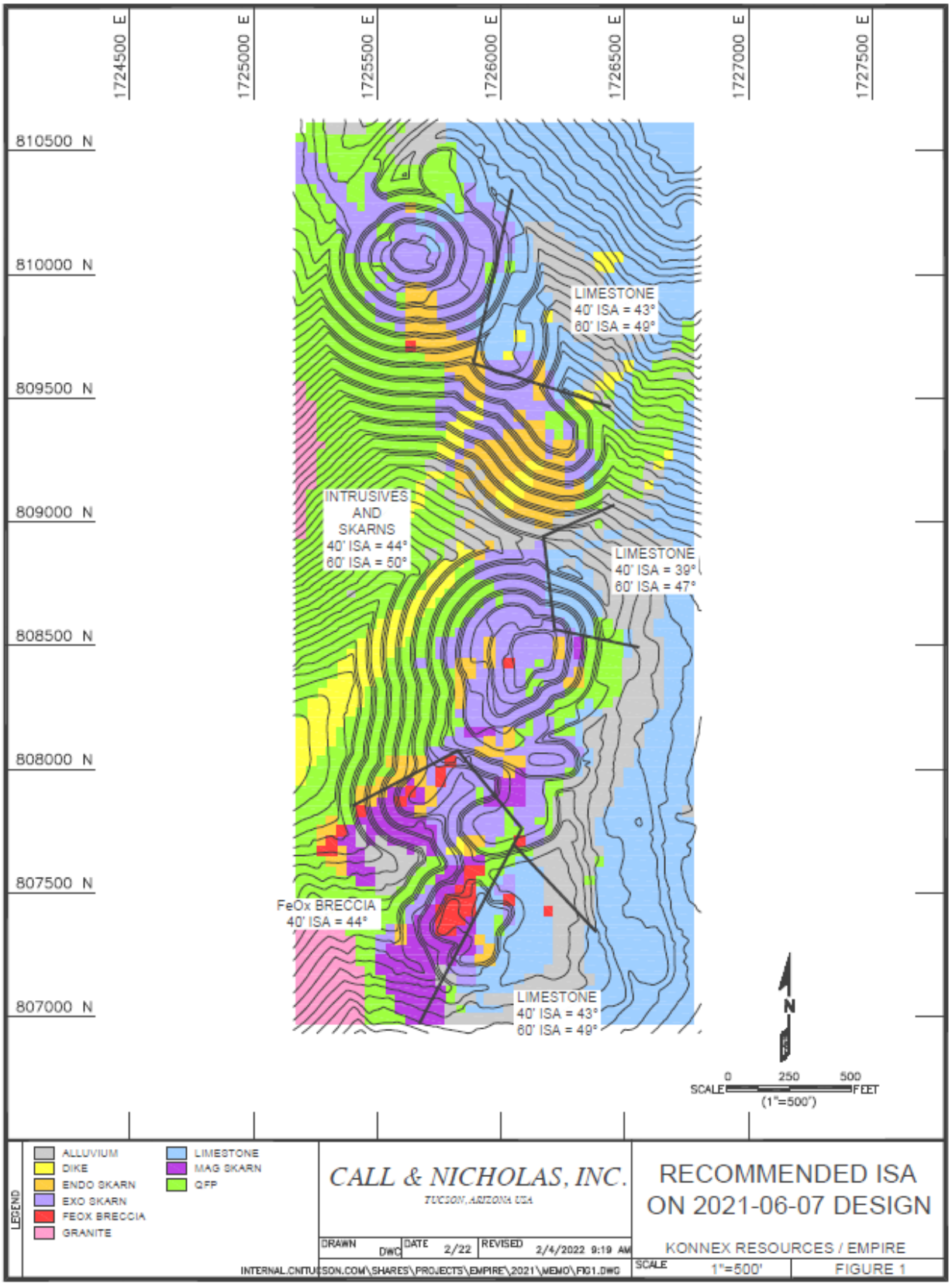


Figure 16-1: Recommend Slope Angles by Sector

16.3 Hydrogeological Considerations

The climate in the vicinity of the Project area is semi-arid, with long snowy winters and short, cool, dry summers. Precipitation occurs largely as spring rainstorms and winter snowfall. Total annual precipitation is measured to be 23.13 total in which includes rainfall of about 9.7 in and approximately 27.6 in of snowfall. The 100-year, 24-hour storm event provided by the National Oceanic and Atmospheric Administration (NOAA) is 2.88 inches.

There are no bulkheads or water discharging from the historic Empire Mine workings, which has approximately 19 miles of interconnected underground workings beneath the patented claims at the Project. Previous to this investigation, groundwater has been reported to have been encountered in only 2 of 445 resource drill holes, both of which are north of the proposed open pit mine. In the first hole groundwater was encountered at 140 ft at 20 gallons per minute, which increased to 40 gpm at 200 ft but the flow dissipated to no discharge within 3 hours. The second hole encountered a trace of groundwater at 400 ft, which increased to 5 gpm at 440 ft and 9 gpm at 500 ft.

Based on the current hydrological understanding and the presence of underground workings that will act as a drain within the open pit area, pit dewatering is expected to be minimal. However, pit dewatering is still anticipated to be required from storm events and spring runoff as the soils and near surface rock will produce high runoff versus infiltration. Drainage diversions ditches will be established around the upper crest of the open pit to divert storm water. Diesel power portable pumps will be used in the pit to capture storm water that will be pumped to settling ponds for use in the ore processing. Both underdrain and surface flows from the waste rock area will be captured in settling ponds prior to either reuse in process or in large events prior to discharge.

16.4 Pit Designs

The Measured and Indicated Mineral Resources were evaluated using a Lerch Grossman pit optimizer to generate optimized pit shells. Pit shells were generated based on varying metal prices and pit optimization parameters for costs, metallurgical recoveries, smelter terms and pit slopes shown in Table 15-2. A total of 51 pit shells were generated to determine optimal break points in the pit phases and the final pit phase for each deposit. The Final pit was limited to a $4.00/ lb CuEq pit shell. The pits were designed into three phases for the Mineral Reserve evaluation with the first phase being based on the $1.52/lb CuEq pit shell and the second phase was based on a $2.00/lb CuEq pit shell.

Haul roads are designed at a width of 56 ft, which provides a safe truck width (11.2 ft wide for 45-ton art. trucks) to running surface width ratio of 1:3 with an additional 13.8 ft for a berm and 8.2 feet for a drainage ditch. Maximum grade of the haul roads is 10%, except for the lower benches where the grade is increased to 13%, and the ramp width is narrowed to 44 ft to minimize excessive waste stripping. The pit design criteria are presented in Table 16-2. The pit designs for all three phases shown in Figure 16-2 through Figure 16-4.

Table 16-2: Pit Design Criteria

Category	Parameter
Ramp Widths	56 ft.
Ramp Grade	10%
Ramp Widths Pit Bottom	44 ft.
Ramp Grade Pit Bottom	13%
Mining Level Heights	20 ft.

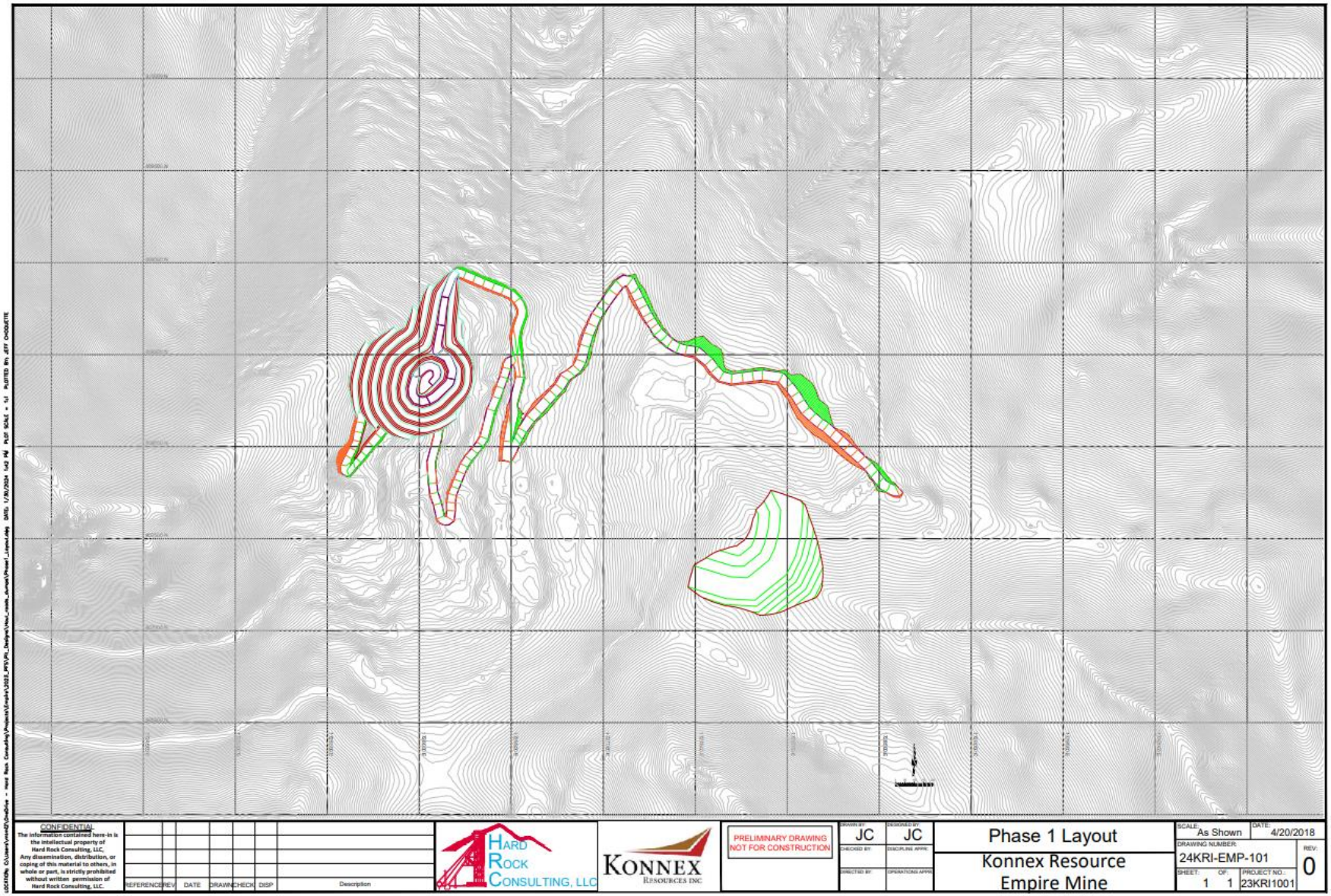


Figure 16-2: Phase 1 Pit Design

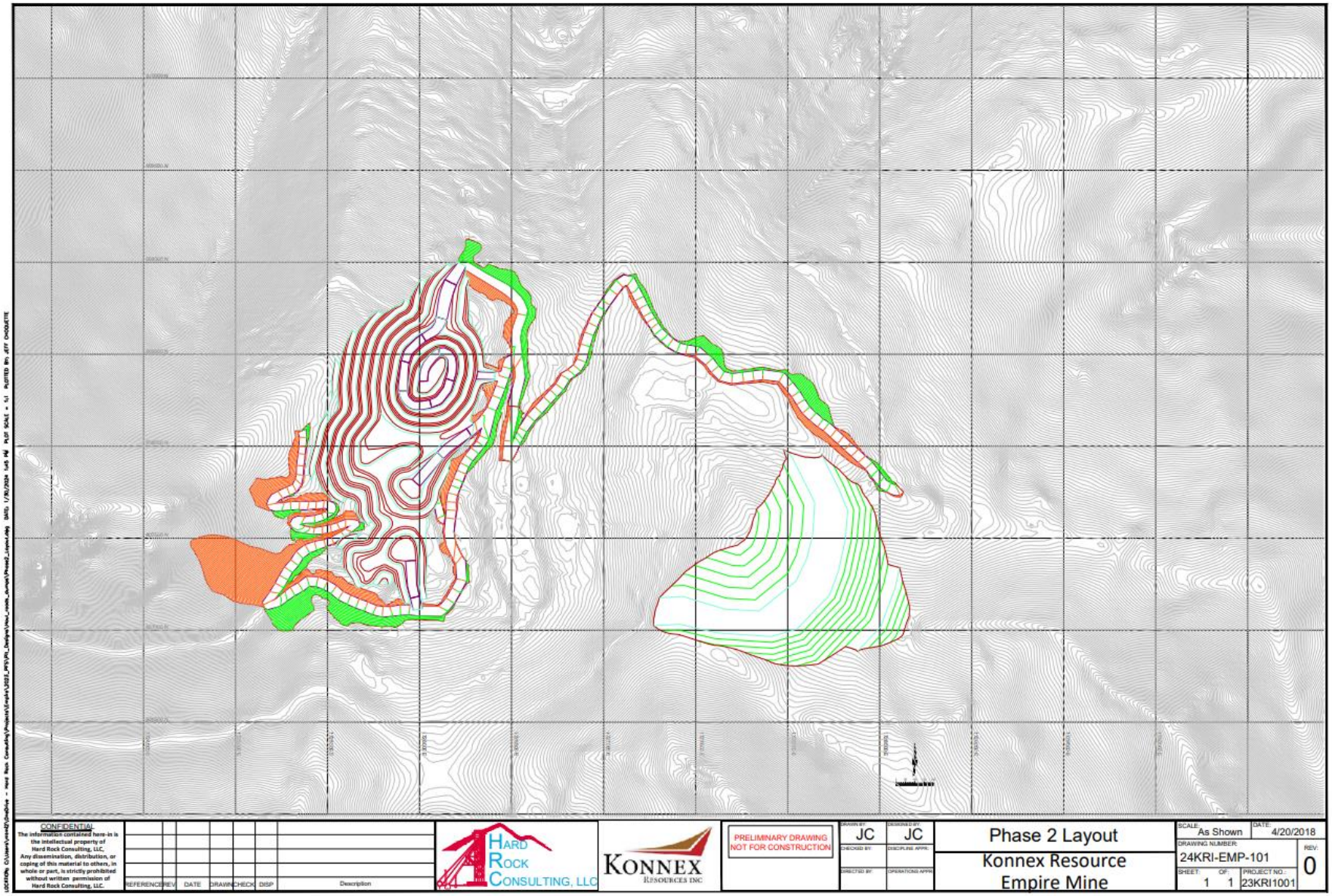


Figure 16-3: Phase 2 Pit Design

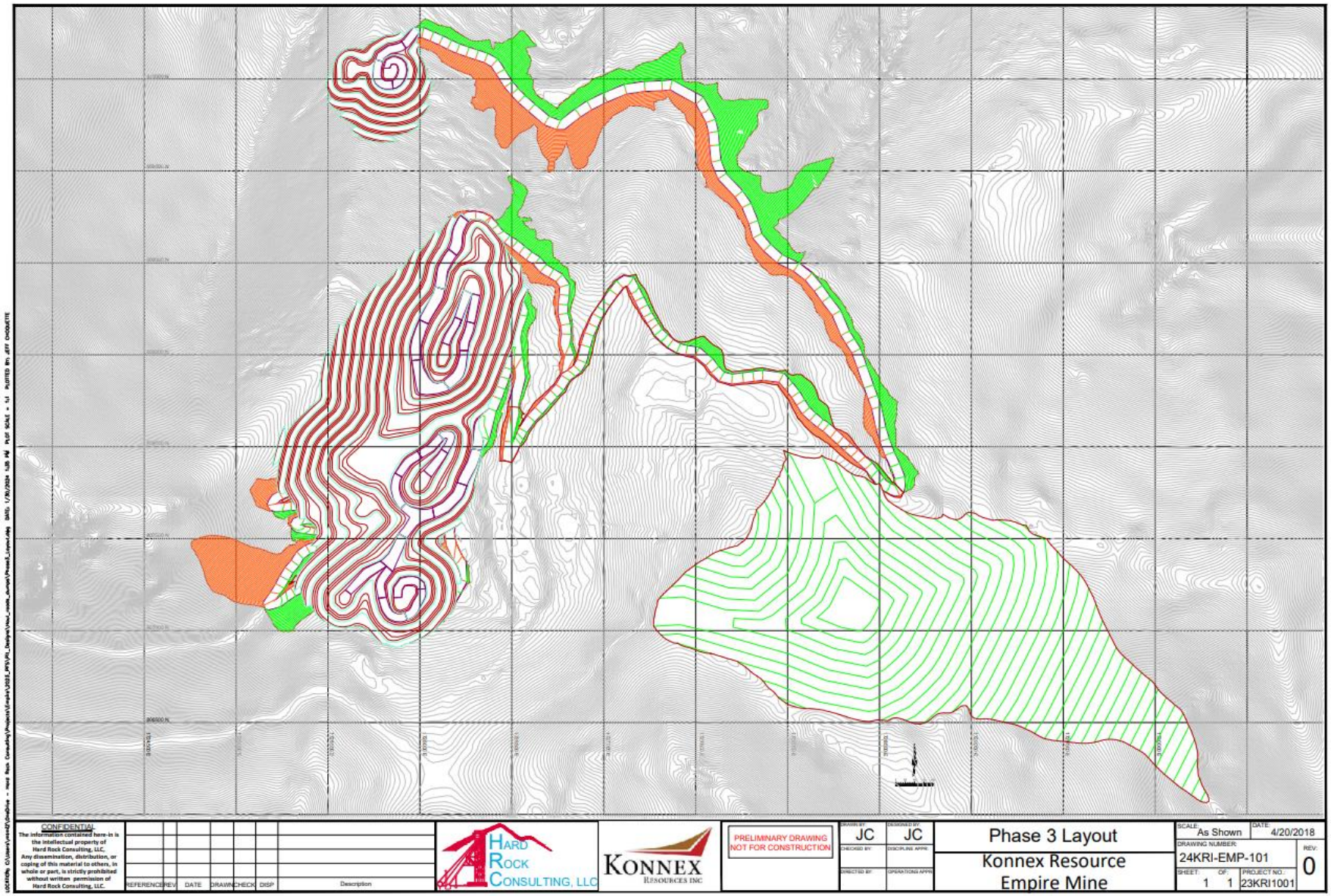


Figure 16-4: Phase 3 Pit Design

16.5 Waste Rock Storage Facility Designs

The Waste Rock Storage Facility (WRSF) is shown above in Figure 16-5. The WRSF was designed with an ultimate capacity of approximately 19 million tons and a 3:1 ultimate slope ratio, aiming to optimize layout and provide a starter embankment for the dry stack tailings facility. The phases of constructing the WRF to optimize the embankment storage are shown in Figure 16-2 and Figure 16-3 along with the pit designs. Lift heights for the WRSF have not been finalized, but a strategy for determining them will be developed in subsequent stages of the project. The haulage profiles included in the mine plan assume that the lifts will be approximately 40 feet.

The WRSF will incorporate a single perforated drainpipe underdrain to intercept ground water flow and direct them to a discharge with a rip rap outlet at the toe to prevent erosion. Additionally, an underdrain measuring 16 ft wide by 3.5 ft deep, enveloped in geotextile to prevent clogging, will traverse the extent of the WRSF. The underdrain serves to facilitate drainage and mitigate saturation within the WRSF. Future design considerations for the WRSF will include slope stability analysis, compaction methods, and adherence to relevant environmental and regulatory standards. Future studies should evaluate the possibility of backfilling the pit in phases to reduce the overall disturbance required for the waste dump.

A low-grade ore stockpile is planned to be located between the open pit and the tailings storage facility as shown in Figure 16-5. The PFS mine plan includes a maximum planned capacity of 867,000 tons during year 7 of the mine plan.

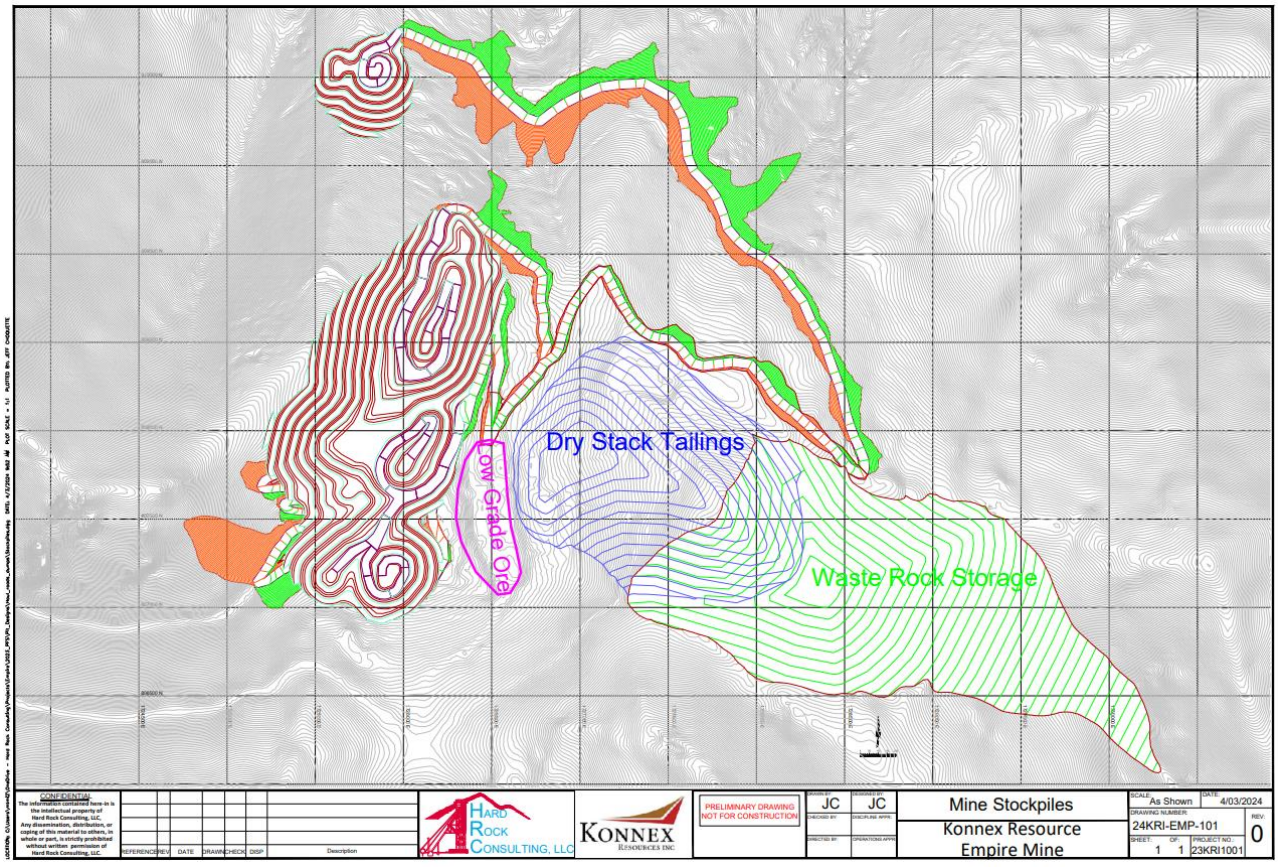


Figure 16-5: Waste Rock Storage Facility and Low Grade Ore Storage Facility

16.6 Dilution and Mining Losses

The mineral resource estimate for Empire is considered to be internally diluted by compositing. HRC also calculated an external diluted grade for all of the grade elements; these diluted grades were used for the mineral reserve calculation. To estimate the external dilution, each side of every ore block was queried to determine if it had a waste block adjacent to each side. If the adjacent block was determined to be waste, then 16.7% of the waste block was included in a weighted average grade estimate for the block. The 16.7% of the waste block is calculated based on a wedge with a 8-foot-wide bottom included as dilution. If two sides of the block are adjacent to waste then the dilution percent is 28.6%, three sides would be 37.5% and all four sides would be 44.4%.

This was applied to all metals and on average, the dilution amounted to 10.3% percentage for the mineral reserve pit design above the $19.00/t NSR. The average grades decreased by approximately 5% and the metal content went up by 4.7% due to the inclusion of the metal content from the dilution material. These percentages were then applied to the $20.50/t NSR cutoff that was used for reporting the final Mineral Reserves.

16.7 Pre-Production Development

The mine preproduction requirements at the project are minimal given the presence of mineable mineralization near the bedrock surface. The majority of the road network is in place from the historic mining operations, but the haulage roads will have to be widened and improved, Figure 16-6 shows the layouts of the haul roads required for the phase 1 operations. The initial roads will provide access to the upper benches of the phase 1 pit and the WRSF. Topsoil will be stockpiled in designated storage areas until it can be reused for on-site reclamation. Waste material from the pit areas may be required for construction of the mill, crusher area and other infrastructure.

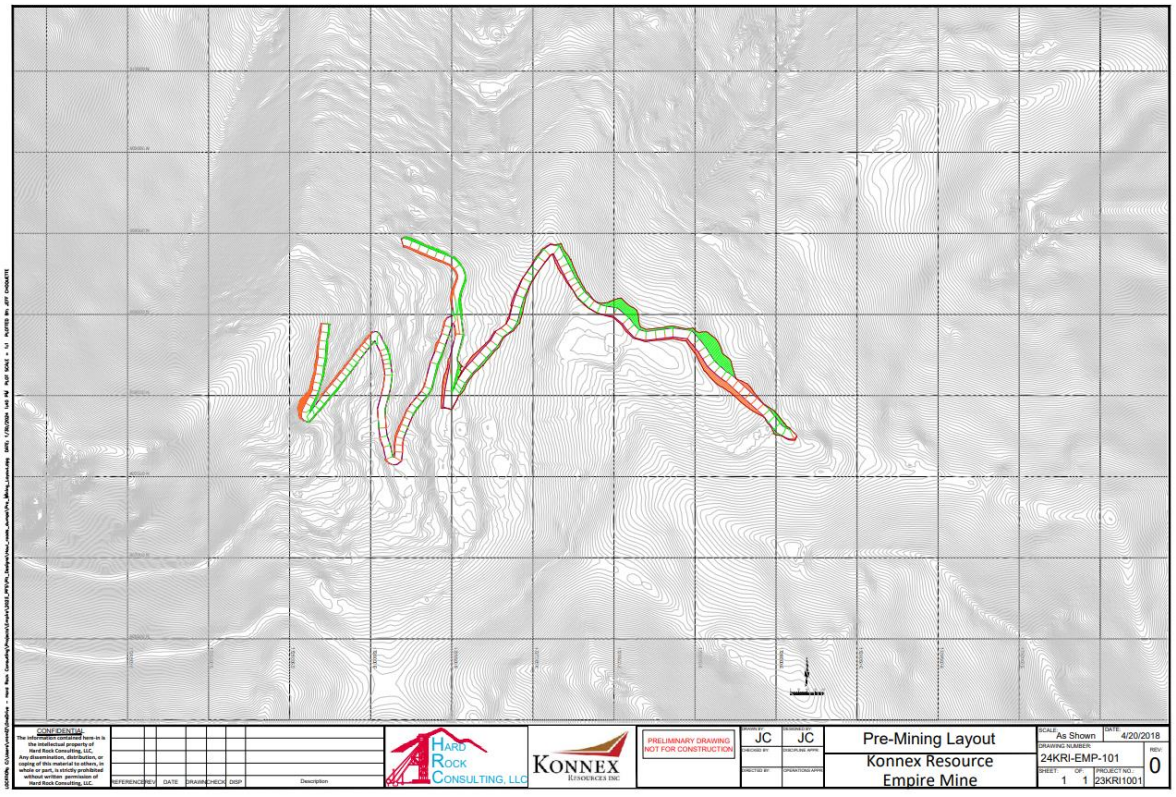


Figure 16-6: Pre-Production Haul Roads

16.8 Mine Production Schedule

Production of mineralized material from the open pit is driven by the nominal ore rate of 4,000 t/d, which is equivalent to 1.4 million tons per annum (average of 350 days per year, or 96% availability) and results in a mine life of approximately eight years. During the mining of pit ore, the peak mineralized material and waste production is estimated at 11,500 t/d and an average production rate of 10,000 t/d. A low grade ore stockpile that reaches a peak capacity of 874,000 tons is planned near the crusher and also between the open pit and tailings storage facility. The stockpile is used to supplement mill feed during high stripping periods of the pit phases. The average life of mine stripping ratio is estimated to be 1.69:1 for the open pit.

The cutoff grade used for the mine schedule is based on the NSR values assigned to the block model described in report Section 15.2.3. The NSR value is based on the diluted metal grades and the dilution approach is described in Section 15.2.2. The cutoff to the low-grade ore stockpiles is $20.50/t NSR and the cutoff for high grade ore is $25.00/t.

The total tons of ore and waste that will be mined during the life of the project are summarized in Table 16-3. The production schedule by pit phase and year are shown in Figure 16-7 and Figure 16-8.

Table 16-3: Yearly Mine Production Schedule

	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Life-of-Mine
All Tons are in 1,000's	**TOTAL**	**TOTAL**	**TOTAL**	**TOTAL**	**TOTAL**	**TOTAL**	**TOTAL**	**TOTAL**	**TOTAL**	**Yr -2 - Yr 8**
MINE PRODUCTION										
Pit High Grade Ore	80	1,254	1,453	1,367	1,173	1,348	1,409	970	687	9,740
Grade: CuT (%)	0.54	0.67	0.59	0.62	0.44	0.42	0.40	0.48	0.69	0.53
Grade: Au (oz/t)	0.005	0.012	0.010	0.007	0.013	0.008	0.009	0.007	0.014	0.010
Grade: Ag (oz/t)	0.73	0.72	0.49	0.45	0.44	0.29	0.25	0.37	0.62	0.44
Pit Low Grade Ore	7	53	89	84	238	313	328	232	46	1,390
Grade: CuT (%)	0.27	0.19	0.24	0.23	0.20	0.21	0.20	0.21	0.12	0.20
Grade: Au (oz/t)	0.001	0.006	0.004	0.004	0.008	0.006	0.008	0.007	0.012	0.007
Grade: Ag (oz/t)	0.38	0.46	0.31	0.28	0.18	0.19	0.18	0.19	0.37	0.22
Pit Waste - Rock	615	1,250	1,238	1,988	2,743	2,391	2,475	3,891	2,215	18,806
Total Tons Mined	**702**	**2,556**	**2,780**	**3,439**	**4,153**	**4,051**	**4,212**	**5,093**	**2,948**	**29,936**
Other, Rehandle etc..	80	268	243	240	448	293	240	664	874	3,350
Total Tons Moved	**782**	**2,825**	**3,024**	**3,679**	**4,601**	**4,344**	**4,452**	**5,757**	**3,822**	**33,287**
Strip Ratio	7.04	0.96	0.80	1.37	1.94	1.44	1.42	3.24	3.02	1.69
Tons ore mined/day	0.2	3.4	4.0	3.7	3.2	3.7	3.9	2.7	1.9	
Total Tons mined/day	1.9	7.0	7.6	9.4	11.4	11.1	11.5	14.0	8.1	
Processed Tons	**0**	**1,362**	**1,400**	**1,404**	**1,400**	**1,400**	**1,400**	**1,404**	**1,360**	**11,130**
Total Cu%	**-**	**0.65**	**0.58**	**0.62**	**0.41**	**0.41**	**0.40**	**0.40**	**0.45**	**0.49**
Total Au oz/t	**-**	**0.011**	**0.009**	**0.008**	**0.012**	**0.008**	**0.009**	**0.007**	**0.010**	**0.009**
Total Ag oz/t	**-**	**0.71**	**0.48**	**0.46**	**0.41**	**0.28**	**0.25**	**0.32**	**0.42**	**0.42**

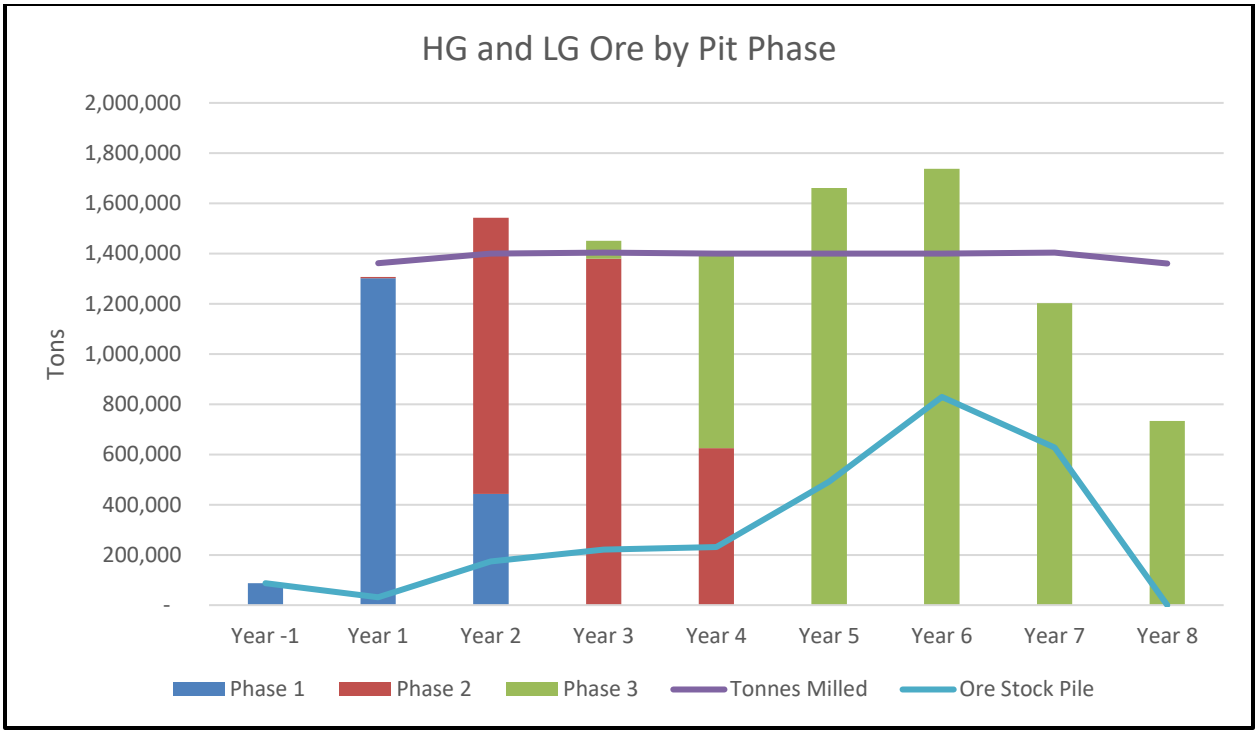


Figure 16-7: Yearly Ore Schedule

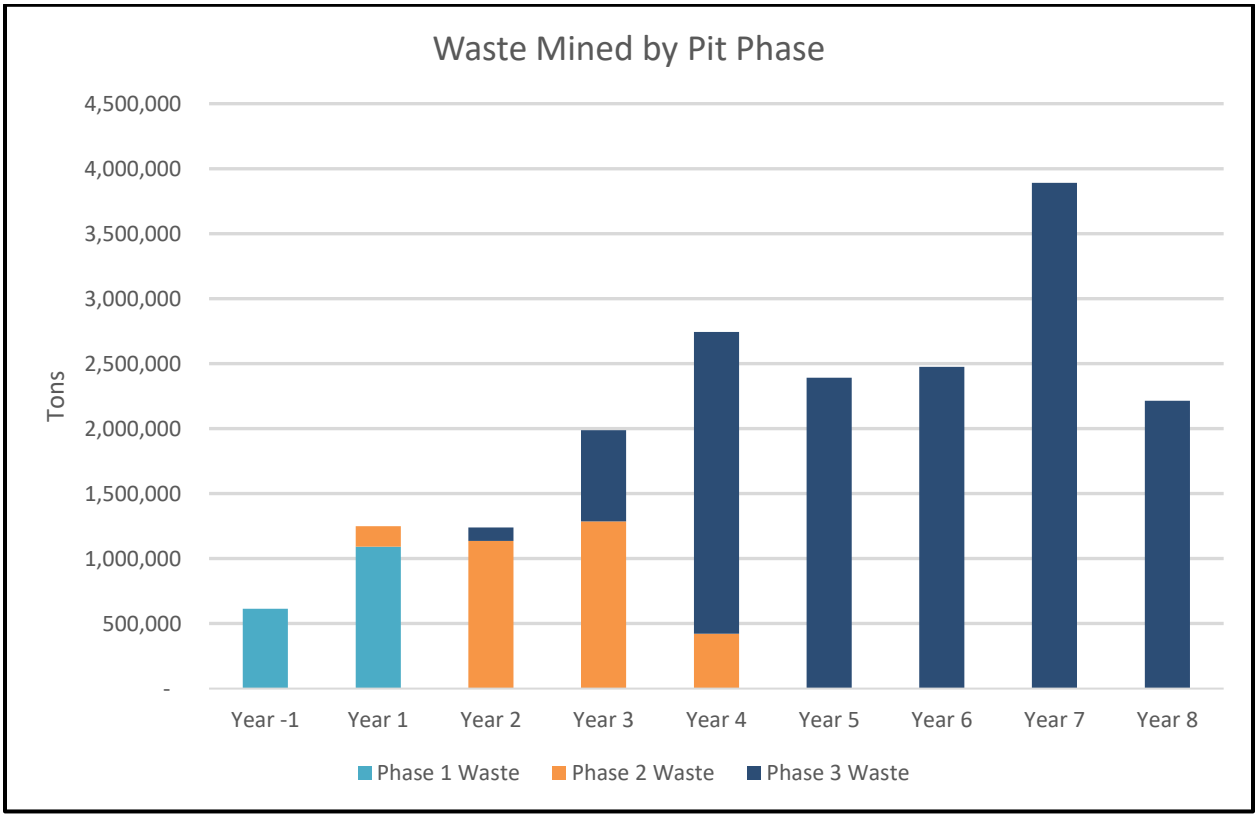


Figure 16-8: Yearly Waste Schedule

16.8.1 Mining Through Underground Workings

The Empire open pit is planned over old underground mine workings. These old workings consist of open and filled stopes, drifts, and raises. Most of all the openings have been surveyed in the past and the locations are known. There is a possibility that some unknown openings or stope boundaries exist so extra care must be taken in operations. The possibility of ground suddenly subsiding and placing people and equipment at risk makes it necessary that all voids and unusual subsidence be treated very carefully. A set of procedures must be developed prior to operations that identify the required safety parameters that must be followed.

The procedures should include parameters on the design of void blast parameters, probe holes to test for voids before and after collapsing, reporting of unknown voids intercepted during drilling. Operating procedures for operating equipment around and over voids that include minimum distances and marking out procedures. Operating procedures for personnel that are operating around or over voids. Reporting and operating procedures for unusual subsidence areas.

16.8.2 Production Schedule Parameters

The mine production schedule is based on a 7 day per week schedule, with two 12 hour shifts per day. There are four crews planned to cover the rotating schedule. Each 12-hour shift contains a half hour down for blasting and miscellaneous delays, a half hour for shift start up and shutdown and a half hour lunch breaks for a total of 10.5 effective working hours. Table 16-4 below shows typical yearly schedule parameters and hours scheduled. A total of 13 weather shifts are included to account for the anticipated lost shifts during the winter months.

Table 16-4: Mine Schedule Parameters

Mine Schedule Parameters	
Crews	4
Shifts/day	2
Hours/shift	12 hr.
Lunch, Breaks, etc.	0.5 hr.
Blasting, Misc.	0.5 hr.
Startup & Shutdown	0.5 hr.
Days/Year	365 days
Less Weather shifts	13 shifts
Scheduled Hours/Year	**8,604**

The amount of equipment required to meet the scheduled tonnages is calculated based on the mine schedule, equipment availabilities, usages and haul and loading times for the equipment. Equipment mechanical physical availabilities start at 92% for the 745 articulated trucks and 982 loaders and 91% for the production drills. For each 6,800 hours of operation, which is a typical year of production with four operating crews, the mechanical physical availabilities decrease by one percent. The use of availability for all of the equipment is calculated at 88% based on the breaks and down time in the schedule parameters. An additional 85% efficiency factor is applied to all of the equipment for calculating the total units of equipment required. Equipment availability parameters are presented in Table 16-5.

Table 16-5: Equipment Availabilities

745 Art. Trucks and 982 Loaders			
Hours	Physical Availability	Use of Availability	Efficiency
0	92%	88%	85%
6,800	91%	88%	85%
13,600	90%	88%	85%
20,400	89%	88%	85%
27,200	88%	88%	85%
34,000	87%	88%	85%
40,800	86%	88%	85%
47,600	85%	88%	85%
54,400	84%	88%	85%
61,200	83%	88%	85%
68,000	82%	88%	85%
74,800	81%	88%	85%
81,600	80%	88%	85%

Production Drills			
Hours	Physical Availability	Use of Availability	Efficiency
0	91%	88%	85%
6,800	90%	88%	85%
13,600	89%	88%	85%
20,400	88%	88%	85%
27,200	87%	88%	85%
34,000	86%	88%	85%
40,800	85%	88%	85%
47,600	84%	88%	85%
54,400	83%	88%	85%
61,200	82%	88%	85%
68,000	81%	88%	85%
74,800	80%	88%	85%
81,600	79%	88%	85%

16.8.3 Drilling and Blast Parameters

The design parameters used to define drill and blast requirements are based on a 6.0-inch blast holes on a 13 ft by 15 ft pattern in the mineralized zones and 6.0-inch blast holes on a 14 ft by 16 ft pattern in the waste zones. Benches will be blasted and mined on 20 ft levels. Buffer rows and pre-shear are planned to allow for controlled blasting and minimize damage to the highwalls. The number of blast holes and blast hole drills required each month or year is calculated based on the parameters presented in Table 16-6 and used in deriving the operating costs. One rotary production drills is required and a smaller track mounted drill is planned for pre-shear drilling, bedrock pioneering and drilling void patterns to collapse the old workings.

Controlled blasting is needed to achieve limit damage to pit slopes. This includes the general use of trim rows, buffer rows, electronic detonators with timings based on signature hole analysis, scaling, free-face blasting, and other blast-damage mitigation measures. To achieve angles greater than 47 to 50 degrees, presplit blasting is planned. A 75-degree average bench face angle must be achieved to meet the 50-degree interramp slope angle. Ledges at interfaces typical of multi-lift bench faces typically reduce the bench-face angle. To eliminate these ledges and minimize damage to the bench face, presplit blasting of the entire 60-foot triple bench is required. This requires drill rigs capable of drilling 4.5-inch diameter holes to 60 feet in depth at an estimated spacing of six feet for all walls exceeding 47 degrees. To avoid excessive blast damage to bench faces, drill rigs for production blasting have been selected based on the burden required for a 20-foot mining increment.

Table 16-6: Drill and Blast Parameters

Production & Wall Control Blast Pattern Data				Production Pattern		Wall Control Pattern		
		Drilling & Blasting Parameters		Ore	Waste			
			Units	Rock	Rock	Buffer	Buffer	Preshear
		Tonnage Factor	t/cubic foot	0.095	0.085	0.085	0.085	0.085
	Blast Pattern Details							
		Bench Height	feet	**20.00**	**20.00**	**20.00**	**20.00**	**20.00**
		Sub Drill	feet	**3.50**	**3.50**	**3.50**	**0.00**	**0.00**
		Diameter of Hole	inches	**6.00**	**6.00**	**6.00**	**6.00**	**4.50**
		Staggered Pattern Spacing	feet	**15.00**	**16.00**	**12.00**	**10.00**	**6.00**
		Staggered Pattern Burden	feet	**13.00**	**14.00**	**14.00**	**12.00**	**6.00**
		Drill Equivalent Square Pattern	feet	14.00	15.00	13.00	11.00	6.00
		Hole Depth	feet	23.50	23.50	23.50	20.00	20.00
		Height of Stemming or Unloaded Length	feet	11.00	12.00	15.00	15.00	
	Material Quantity							
		Volume Blasted/Hole	cubic feet	3,920	4,500	3,380	2,420	720
		Tons Blasted/Hole	tons	372	383	287	206	61
	Powder Factor							
		Percent Emulsion		30%	30%	30%	30%	30%
		Percent Anfro		70%	70%	70%	70%	70%
		Density of Powder	lbs/ft3	**71.79**	**71.79**	**71.79**	**71.79**	
		Loading Density	lb/ft	14.10	14.10	14.10	14.10	5.00
		Powder/hole	lb/hole	176.20	162.11	119.82	70.48	5.51
		Powder Factor	lb/t	**0.473**	**0.424**	**0.417**	**0.343**	**0.090**
		Powder Factor	lb/ft3	0.045	0.036	0.035	0.029	0.008
Drill Productivities								
	Penetration Rate		ft/hr	165.00	165.00	165.00	165.00	120.00
		Penetration Rate	ft/min	**2.75**	**2.75**	**2.75**	**2.75**	**2.00**
	Cycle Time Estimate							
		Drilling Time	minutes	8.55	8.55	8.55	7.27	10.00
		Steel Handling Time	minutes	0.00	0.00	0.00	0.00	0.50
		Set up Time	minutes	1.75	1.75	1.75	1.75	2.00
		Add Steel	minutes	0.00	0.00	0.00	0.00	2.00
		Pull Rods	minutes	0.50	0.50	0.50	0.50	2.00
		Total	**minutes**	**10.80**	**10.80**	**10.80**	**9.52**	**16.50**
Drilling Factors for Wall Control								
		Buffer Holes - 2 Rows						
	Wall Control Drill Holes Required							
		Buffer Holes - 2 Rows	holes/feet		**0.17**			
		Material to Remove from Production Blast	tons/feet		**51.00**			

16.8.4 Load and Haul Parameters

The design parameters used to define the loading and hauling requirements are shown in Table 16-7. The method of material transport at the Project is expected to be open pit mining using two 7.5-yd3 front end loaders as the main loading units. The open pit mineralized material is loaded into 45-ton articulated haul trucks and transported to the crushing site or low-grade stockpile; waste material will be transported to the designated rock waste dump. Haulage profiles for the mineralized material and waste material from each pit phase were generated and used to calculate the truck cycle times, which were then used in the equipment requirement and operating costs calculations.

Table 16-7: Load and Haul Parameters

Loading & Truck Match Calculation		982 Loader Waste	982 Loader Ore
		745 Truck	745 Truck
Bucket Capacity (heaped)	yd3	7.50	7.50
Bank Material Weight Dry	tons/bcy dry	2.30	2.57
Bank Material Weight Wet	tons/bcy wet	2.35	2.63
Bulk Factor (Swell Factor)		1.30	1.30
Loose Material Weight Dry	t/lcy dry	1.77	1.97
% Moisture		2.5%	2.5%
Bucket Fill Factor		0.85	0.85
Effective Bucket Capacity	yd3	6.38	6.38
Wet Material Weight (LCM)	wt/lcy	1.81	2.02
Dry Material Weight (LCM)	dt/lcy	1.77	1.97
Tons/Pass	wt	11.54	12.89
Truck Size Capacity (volume)	yd3 heaped	32.7	32.7
Truck Size Capacity (Tons)	wt	45.2	45.2
Theoretical Passes (volume)	passes	5.13	5.13
Theoretical Passes (Tons)	passes	3.92	3.51
Actual Passes	passes	**4.0**	**4.0**
Truck Load - Volume (volume)	yd3	25.5	25.5
Truck Load - Volume (Tons)	wt	46.1	51.6
Truck Load for Productivity	dt	45.0	50.3
Truck Capacity Utilized (Tons)	by weight	102.1%	114.1%
Truck Capacity Utilized (volume)	by volume	78.0%	78.0%
Average Cycle Time	sec	50	50
Truck Spot Time	sec	45	45
Load Time per Truck	sec	245	245
Load Time per Truck	minutes	4.08	4.08
Maximum Productivity	trucks/hr	14.7	14.7
Insitu Volume/Hour	bcy/hr	288.2	288.2
Tons/Hour	**dt/hr**	**661.5**	**739.3**

16.9 Mining Equipment

For the planned open pit mining operation, the amount of equipment required to meet the scheduled tonnages is calculated based on the mine schedule, equipment availabilities, usages of availability, efficiency factors, and haul and loading times for the equipment.

The mine production equipment is expected to include two 7.5yd^3 front-end loaders (982 Cat) as the main loading units. Initially only one loader is required and a second is added in year three of production. The haulage is planned to be 45-ton articulated haul trucks (745 Cat) which will initial require the truck to meet production and increase to a total of six trucks during the stripping of the phase 3 pit. 60-ton fixed frame haul trucks were also evaluated as part of the study, but the lower costs and maneuverability made the articulated trucks the preferred choice. Only one production drill is required to meet the schedule, but a pre-shear drill is also planned for wall control purposes and backup drilling. Table 16-8 lists the total mine production equipment requirements.

Table 16-8: Mine Production Equipment

Description	Initial Units	Additional Units	Replacement Units	Total Units
982 Loader	1	1	0	2
Production Drill	1	0	0	1
Pre-Shear Drill	1	0	0	1
Haul Truck - 745	3	3	0	6

Support equipment consists of one Cat D8 dozers and one D9 dozer. A 16 ft road grader will service the haul roads along with a 9,000-gallon water truck. Six mobile light plants will be required for lighting the working areas during night-time production. A maintenance service truck with a mobile crane will be needed for field maintenance and a self-contained fuel lube truck will be needed for infield fueling. The anticipated mine support equipment is listed in Table 16-9.

Table 16-9: Mine Support Equipment

Description	Initial Units	Additional Units	Replacement Units	Total Units
16' Grader	1	0	0	1
Water Truck	1	0	0	1
D9 Dozer	1	0	0	1
D8 Dozer	1	0	0	1
Lube/Fuel Trk	1	0	1	2
Service Truck	1	0	1	2
Skid Steer	1	0	1	2
Pickups mine G&A	2	0	0	2
Pickups Drilling	1	0	0	1
Pickups Blasting	1	0	0	1
Pickups Maintenance	2	0	1	3
Pickups Engineering	2	0	0	2
Pickups Geology	1	0	0	1
Light Plants	6	0	1	7
Genset	1	0	0	1
Compressor	1	0	1	2
Dump Truck	1	0	0	1
Excavator	1	0	0	1
Dewatering Pump	1	0	0	1
Welders	2	0	1	3

17. RECOVERY METHODS

The Empire Project is identified as a 4,000 stpd oxide plant as presented in this section.

The key criteria used in the process design of the Process Plant have been largely derived from metallurgical test work and, where appropriate, have been provided by Konnex, or based on Forte experience. The design criteria and flowsheet development are discussed in this section.

17.1 Process Design Criteria

Based on initial metallurgical test work, preliminary mineral resources, and various necessary assumptions, a process design criteria were developed for the Empire copper oxide plant. Table 17-1 lists the highlights of operating parameters, equipment throughputs, solution streams, and copper output.

Table 17-1: Project Design Criteria

Item	Units	Nominal	Reference
General Site Location			
Mill Location	NAD 83 Idaho State Plane, Central Zone, US foot	N:808600 E:1726890	Forte
Mill Elevation	ft	8380	Forte
Max. Summer Temperature	°F	104	Forte
Min. Winter Temperature	°F	-33	Forte
Annual Precipitation	in/yr	21.6	Forte
Pan Evaporation average	In/yr	41	Forte
Ultimate Wind Speed	mph	115	ASCE 7-10
Frost Depth	in	36	Forte
Snow Load> 6300	psf	128	U of I
General Project Information			
Operating Days per Year	d	350	Client
Operating Hours per Day	hr	20 (2-10 hr shifts)	Client
Operating Hours per Year	hr	7,000	Calculated
Estimated Mechanical Availability	%	92	Standard
Actual Operating Hours per Year	hr	6,440	Calculated
Plant Capacity	stpd	4,000	Client
Annual Ore Process per Year	Mst	1.4	Calculated
Estimated Project Life	year	9	
Plant Feed Grade Estimates			
Average Cu Grade	%	0.5	HRC
Max. Cu Grade	%	0.7	HRC
Min. Cu Grade	%	0.3	
Average Au Grade	oz/st	0.01	
Average Ag Grade	oz/st	0.41	
Design ROM Ore Bulk Density	lb/ft3	125	Assumed
Design ROM Ore Dry Solids Specific Gravity	unit	2.8	
110 - Primary Crusher			
Design Feed Size Distribution F_{100} F_{80}	 in in	 10 4	Fragmentation Report

Item	Units	Nominal	Reference
Bond Crushing Work Index	kWh/st	14.3	F.A. Lab
Bond Abrasion Index	g	0.0773	F.A. Lab
Design Feed Rate	st/hr	220	Calculated
Crusher Type		Jaw	Assumed
Crusher Size	in x in	30 x 40	Vendor (Nordberg)
Crusher HP	Hp	150	Vendor (Nordberg)
Design Open Side Setting	in	4	Handbook
Design Primary Crusher Product Size Distribution P_{100} P_{80}	 in in	 5 2.5	Handbook
120 - Secondary Crusher			
Design Feed Rate	dstph	220	Calculated
Mobile vs Fixed		Fixed	
Crusher Type		Medium Cavity Cone	Vendor (Nordberg)
Crusher Size	ft	4	
Crusher HP	Hp	300	Mining Cost Service
Number of Crushers		1	Standard
Design Open Side Setting	in	2.5	Handbook
Design Discharge Size Distribution P_{100} P_{80}	 in in	 1.25 5/8	Handbook
130 - Crushing Plant Screen			
Screen Area Required	ft^2	80	Calculated
Screen Type		Horizontal	
Screen Size	ft x ft	6 x 16	
Number Screen Decks		1	Forte
Screen Deck Opening Size	in	3/4	Assumed
140 - Tertiary Crusher			
Number of Crushers		1	Forte Assumed
Design Feed Rate	dstph	440	Vendor
Crusher Type		Short Head Cone	Forte
Crusher Size	ft	3	Vendor
Crusher HP	Hp	200	Mining Cost Service
Design Open Side Setting	in	1	Forte
Design Product Size Distribution P_{80}	 in	 1/2	Standard
150 - Fine Ore Bin			
Storage	hrs	16	Assumed
Capacity	tons	3,520	Calculated
Size (diam x H)	ft	36 x 57	Calculated
Volume	ft^3	58,000	Calculated
210 - Milling (Ball Mill) Circuit			
Design Feed Size - F_{80}	in	1/2	Assumed
Ball Mill Classification Size - P_{80}	microns	150	F.A. Lab
Ball Mill Size – Diameter x EGL	ft	14 x 20	Vendor
Circulating Load	%	250	Standard
Power	Hp	2,250	Calculated
Speed Control	Type	Soft Start	Assume
Speed – Fraction Critical Speed	%	75	Standard
Ball Charge	%	40	Standard

Item	Units	Nominal	Reference
Largest (Feed) Ball Size	in	3	Assumed
Bond Ball Mill Work Index	kWh/st	15.5	F.A. Lab
Bond Abrasion Index	g	0.0773	F.A. Lab
Ball Mill Discharge Density-% Solids	%	65	Assumed
Ball Mill Discharge-flowrate	gpm	2,418	Calculated
Cyclone Feed Density – wt. % solids	%	52	Calculated
Cyclone Feed Flowrate	gpm	3,300	Calculated
Cyclone Overflow – Slurry Density	%	35	Assumed
Cyclone Overflow – Solids Flowrate	t/hr.	175	Calculated
Cyclone Overflow – Slurry Flowrate	gpm	1560	Calculated
Cyclone Manufacturer		Krebs	Assumed
Cyclone Size/Model		20	Vendor
Number of Cyclones – Operating/Spare		3+1	Forte
Cyclone Operating Pressure	PSI	10	Handbook
310 - Rougher Flotation Circuit			
Feed Flow Rate @ 35% Solids	ft^3/min	210	Calculated
Conditioning Tank Residence Time	min	2	Assumed
Conditioning Tank Volume Required	ft3	420	Calculated
Conditioning Tank Size – Dia x H	ft	9 x 8	Calculated
Rougher Flotation Residence Time - Lab	min	9	F.A. Lab
Lab to Plant Scale-Up Factor		2.25	Standard
Rougher Flotation Residence Time - Plant	Min	20	Calculated
Rougher Flotation Bank – Type		Tank	Assumed
Rougher Flotation Bank Total Volume	ft^3	4,200	
Number of Cells	ea	4	
Cell Dimensions L x W x H or Dia x H	ft	11 x 11	Calculated
Rougher Concentrate Mass Pull	%	0.5	F.A. Lab
Rougher Concentrate % Solids	%	20	Assumed
Rougher Concentrate Solids Flowrate	st/hr.	5.8	Calculated
Rougher Concentrate Slurry Flowrate	gpm	105	Calculated
Rougher Tailings % Solids	%	36	
Rougher Tailings Flowrate	ft^3/min	196	Calculated
Rougher Tailings Solids Flowrate	st/hr.	161	Calculated
315 - Carbonate Flotation Circuit			
Feed Flow Rate @ 35% Solids	ft^3/min	210	Calculated
Conditioning Tank Residence Time	min	2	Assumed
Conditioning Tank Volume Required	ft^3	420	Calculated
Conditioning Tank Size – Dia x H	ft	9 x 8	Calculated
Carbonate Flotation Residence Time - Lab	min	4.5	Assumed
Lab to Plant Scale-Up Factor		2.25	Standard
Carbonate Flotation Residence Time - Plant	min	10	Assumed
Carbonate Flotation Bank – Type		Tank	Assumed
Carbonate Flotation Bank Total Volume	ft^3	2,100	
Number of Cells	each	2	
Cell Dimensions L x W x H or Dia x H	ft	11 x 11	Calculated
Carbonate Concentrate Mass Pull	%	0.5	F.A. Lab
320 - First Stage Cleaner Flotation			
First Stage Cleaner Flotation Residence Time – Lab	min	6	F.A. Lab
Lab to Plant Scale-Up Factor		2.25	Standard

Item	Units	Nominal	Reference
First Stage Cleaner Flotation Residence Time - Plant	min	14	Calculated
First Stage Cleaner Flotation Bank Total Volume	ft^3	280	Calculated
Number of Cells		3	Standard
Nominal Volume Per Cell	ft^3	100	Quoted
First Stage Cleaner Concentrate % Solids	%	15	Forte
First Stage Cleaner Concentrate Flowrate	ft^3/min	3.8	Calculated
First Stage Cleaner Concentrate Solids Flowrate	st/hr.	1.2	Calculated
First Stage Cleaner Tailings % Solids	%	22	Calculated
First Stage Cleaner Tailings Flowrate	ft^3/hr.	10.2	Calculated
First Stage Cleaner Tailings Solids Flowrate	t/hr.	4.7	Calculated
325 - Second Stage Cleaner Flotation			
Second Stage Cleaner Flotation Residence Time – Lab	min	2	F.A. Lab
Lab to Plant Scale-Up Factor		2.25	Standard
Second Stage Cleaner Flotation Residence Time - Plant	min	4.5	Calculated
Second Stage Cleaner Flotation Bank Total Volume	ft^3	100	Assumed
Number of Cells		1	Standard
Nominal Volume Per Cell	ft^3	100	Assumed
330-Cleaner Concentrate Thickener (Additional Test Work Required)			
Feed Solids	%	15	Calculated
Solids Feed Rate	t/hr.	1.2	Calculated
Underflow Slurry Density (% Solids)	%	65	Assumed
Design Thickener Area	ft^2	28 (33.6)	Assumed
Selected Thickener Diameter	ft	6 (6'-8")	Calculated
Selected Thickener Area	ft^2	28.3 (35)	Calculated
Thickener Underflow Flowrate	ft^3/min	0.64	Calculated
340-Concentrate Filter (Additional Test Work Required)			
Filter Feed Rate (Thickener Underflow)	ft^3/min	0.64	Calculated
Filter Feed Slurry Density	Wt. % solids	55	Assumed
Filter Type		Drum	Forte
Filter Cake Target Moisture	%	20	Assumed
Filter Cake Production	T/hr.	1.3	Calculated
Filter Size Dia x Length	ft x ft	6x4	Handbook
Filter effective area		76	Handbook
Filter Cake Production	ft^3/hr	30	Calculated
Filtrate Flowrate	ft^3/min	3.1	Calculated
340-Roll Away Bin			
Dry Concentrate Storage Capacity	st	24	Assumed
Dry Concentrate Storage Capacity	ft^3	486	Assumed
Number of Bins		6	Assumed
Dry Concentrate Bin Size L x W x H	ft	20x7x4	Assumed
410-Leach Circuit Feed Thickener (Additional Test Work Required)			
Feed Slurry Solids	%	36	Calculated
Solids Feed Rate	st/hr.	161	Calculated
Underflow Slurry Density (% Solids)	%	65	Assumed
Settling Rate (Unit Area)	ft^2/t/d	1	Assumed (Handbook)
Minimum Thickener Area	ft^2	3864	Calculated
Design Thickener Area	ft^2	3864	Calculated

Item	Units	Nominal	Reference
Calculated Thickener Diameter	ft	70.1	Calculated
Selected Thickener Diameter	ft	70	Calculated
Selected Thickener Area	ft^2	3864	Calculated
Thickener Underflow Flowrate	ft^3/min.	113.2	Calculated
420 - Copper Leaching Circuit			
Solids Feed Rate	st/hr.	161	Calculated
Feed Slurry Fraction Solids	%	45	Forte
Feed Slurry Volumetric Feed Rate	ft^3/min	148	Calculated
Design Leaching Residence Time	hrs.	5	Forte
Total Leach Volume required	ft^3	46,500	Calculated
Number of Leach Tanks		3	Forte
Individual Leach Tank Volume Required	ft^3	15,500	Calculated
Calculated Tank Dimensions – Dia x H	ft	26.5 x 28	Calculated
510 - Leach (Final) Tailings Thickener (May Not be Necessary – Additional Test Work Required)			
Feed Solids	%	44.4	Calculated
Solids Feed Rate	t/hr.	157	Calculated
520 - Leach (Final) Tailings Filter System (More Test Work Required)			
Filter Feed Slurry Density	Wt. % solids	44.4	Calculated
Filter Type		Disc	Purchased
Number of Filters		2	Purchased
Number of Discs per Filter		8	Purchased
Filter Disc Diameter	ft - in	8 - 10	Purchased
Filter Area, Each	ft^2	900	Calculated
Filter Cake Target Moisture	%	15	Assumed
Number of Vacuum Stations		2	Purchased
Vacuum Pump Motor Horsepower	HP	200	
Re-Pulp Tank Residence Time	min	2	Assumed
Re-Pulp Tank Volume Required	ft^3	420	Calculated
Re-Pulp Tank Size – Dia x H	ft	9 x 8	Calculated
610 - Copper Cementation			
Solution Average Feed Rate	ft^3/hr.	863	Calculated
Solution Copper Tenor	g/l Cu	2.26	Calculated
Copper in Feed	lb/hr-st/day	975 - 11.7	Calculated
Cementation Reactor Type		Cone-Tank	Forte
Cementation Residence Time, per tank	min	15	Forte
Number of Reactors Online+Spare		2+1	Forte
Cementation Tank Volume	ft^3	1730	Calculated
Cementation Vessel Size – Dia x H	ft	10x17	Calculated
Wetted Materials of Construction		Stainless Steel	Forte
Iron Scrap Type		Shredded	Forte
Iron Scrap Consumption	t/hr	1463	Calculated
Cementation Underflow Slurry Solids	%	80	Assumed
Cementation Underflow Solids Flow	lb/hr.	926	Calculated
Avg. Cementation Underflow Flowrate	gpm	0.65	Calculated
Solids Discharge Moisture	%	90	Forte

17.2 Flow Sheet Development & Description

The Empire Project employs a three-stage crusher circuit to prepare the ore for grinding in a closed-circuit ball mill, aiming to achieve a particle size of 150 microns (P_{80}). To close the circuit, the ball mill discharge is classified by a cyclone pack. The overflow from the cyclones is directed to a rougher sulfide flotation circuit. Cyclone underflow flows back to the mill feed.

Rougher sulfide concentrate from the rougher-flotation circuit flows to a two-stage cleaner-flotation circuit. The first stage cleaner flotation concentrate is directed to the second stage cleaner flotation circuit. The second stage cleaner concentrate (the final sulfide flotation concentrate) is directed to a thickener. Thickener underflow is then sent to a drum filter, producing a dewatered, salable concentrate containing copper, gold, and silver. The tailings from the first- and second-stage cleaner-flotation banks are returned to the grinding circuit.

Tailings from the rougher flotation circuit are fed to a carbonate flotation circuit to remove acid consuming materials from that stream. The carbonate flotation concentrate is sent to the water treatment plant (WTP) to treat acid effluent form the process. The carbonate flotation tailings are fed to a pre-leach thickener which removes water from the slurry to reduce the volume of slurry flowing to the agitated leaching circuit.

The thickened underflow is fed to a three-stage agitated, acid leach circuit. After leaching, the leach slurry is filtered and washed using a two-stage disc filter circuit with an intermediate repulping stage to recover pregnant liquor. In this circuit, the leach slurry is fed to the first disc vacuum filter which recovers the bulk of the pregnant liquor. The filter cake is then mixed with wash water in the repulp tank. The reconstituted slurry is then fed to the second disc filter for final dewatering.

The pregnant and wash liquor filtrates from the two-disc filters flow to cementation tanks, where it is contacted with scrap iron. A reaction between the copper in solution and the solid shredded steel produces a final cement copper product.

The leach residue filter cake from the belt filter will be stored in a designated lined dry stacked tailings storage facility. A portion of the carbonate flotation concentrate may be used to neutralize acid to allow excess solution from the leach circuit to be used as process water. The remainder will be fed to the pug mill where it will be mixed with the leach residue filter cake.

This process is presented schematically and is described in further detail in the following sections. The full set can also be found in Appendix A. 2.

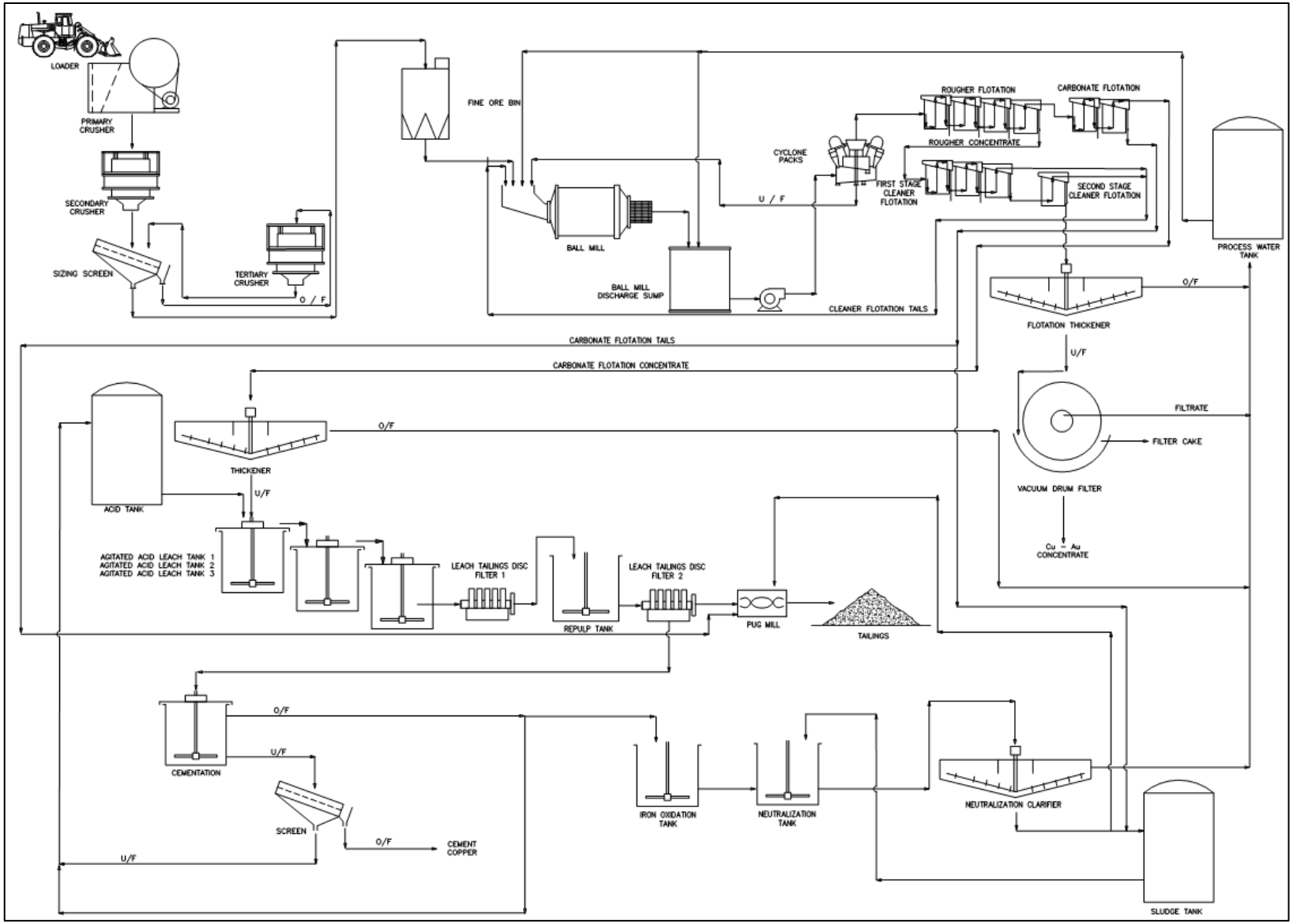


Figure 17-1: Overall Process Flowsheet

17.3 Description of Process Areas

17.3.1 Crushing and Screening

Run-of-mine ("ROM") ore will be delivered to a modular crushing screening plant at a rate of 220 short tons per hour. The ROM material is expected to have a P_{100} of 10" and P_{80} of 4". The crushing-screening plant will consist of a vibrating grizzly feeder, a jaw crusher, a magnet for the removal of tramp iron, a secondary cone crusher, a horizontal screen, and a tertiary cone crusher. The secondary and tertiary crushers operate in closed circuit with the vibrating screen. These components are connected by a series of conveyors. The crushing circuit will produce a final crushed product with a P_{80} of ½", which subsequently feeds the fine ore bin and ball mill circuit.

Secondary and tertiary crushing and screening components will be covered by small open-sided buildings to prevent snow and precipitation from clogging the screens, cone crushers and chutes.

17.3.2 Milling (Ball Mill Circuit)

The ore from the fine ore bin will be drawn down with a belt feeder and then transferred onto the ball mill feed conveyor. A used ball mill has been procured by the Konnex team for the use on this project. The ball mill is an Allis Chalmers 14.5' x 28' (4.42m x 8.53m) grinding ball mill with a single 3,000 HP (2,250 kW) motor. The ball mill will handle a standard recirculating load of 250%, producing a discharge of 588 st/hr at 65% solids. The ball mill discharge slurry will be pumped to Cyclones where the underflow will flow by gravity back to the ball mill feed. The Cyclone bank overflow will be pumped to the rougher flotation circuit at solids flow rate of 167 st/hr and 35% solids.

17.3.3 Flotation Circuit

The cyclone overflow will be sent to rougher flotation which will float copper, gold, silver, and other sulfides. The slurry will be conditioned for 2 minutes before being pumped to the rougher flotation cells which has a residence time of 20 minutes. The rougher flotation bank will be four tank cells of 4,200 ft^3 volume. The rougher concentrate will be upgraded to salable concentration in a two-stage cleaner-flotation circuit, while the rougher tailings will be pumped at 161 st/hr to the carbonate flotation circuit.

Tailings from the rougher flotation circuit are fed into the carbonate flotation circuit. The slurry will be conditioned for 2 minutes before being pumped to the carbonate flotation cells for 10 minutes. The carbonate flotation bank will be two tank cells of 2,100 ft^3 volume. The carbonate concentrate will be pumped to the WTP to treat acid effluent from the process. The carbonate tailings will be pumped to the acid leach circuit.

The cleaner-flotation circuit will consist of two stages. The first-stage cleaner flotation circuit will have three flotation cells in series and the second-stage cleaner flotation circuit will have a single flotation cell. The first-stage cleaner concentrate will be pumped to the second-stage cleaner-flotation circuit. The second-stage cleaner-flotation concentrate will be pumped to the cleaner-concentrate thickener at feed rate of 1.2 st/hr. The underflow of the thickener will be pumped to a vacuum drum filter which will produce an 85% solids filter cake. The filter cake from the drum filter will be carried by a conveyor to a roll away bin where it will be shipped offsite for further processing. The tailings from both stages of the cleaner flotation will be pumped back to the ball mill feed at a flowrate of 10.2 ft^3/min.

17.3.4 Acid Leach Circuit, Disc Filters, and Pug Mill

The carbonate tailings slurry will be pumped at 196 ft^3/min to the leach circuit thickener. Overflow from the thickener will flow to the process water tank while underflow at 55% solids will be pumped to the acid leach circuit. Recycled acid solution from the copper cementation circuit will be added to dilute the leach slurry to 45% solids.

Three agitated leach tanks containing diluted H_2SO_4 will make up the leach circuit. The first tank overflow will gravity feed to the second tank, the overflow of which will then be pumped to the third tank. The slurry will be leached in these tanks for a total of 5 hours.

The leach slurry is filtered and washed using a two-stage disc filter circuit with an intermediate repulping stage to recover pregnant liquor. In this circuit, the leach slurry is fed to the first disc vacuum filter which recovers the bulk of the pregnant liquor. The filter cake is then mixed with wash water in the repulp tank. The reconstituted slurry is then fed to the second disc filter for final dewatering. The disc filters will be fed slurry at 157 st/hr at 44.4% solids. The combined filtrate will be pumped to the cementation circuit.

The washed filter cake will be carried via a conveyor to a PUG mill where concentrate from the carbonate flotation circuit, lime, and possibly cement will be added. A conveyor system will carry the product to a radial stacker where the tailings will be stacked in a dry stacked tailings storage facility (TSF).

17.3.5 Copper Cementation Circuit

Filtrate containing a calculated 2.26 g/L copper will be pumped to the cementation surge tank before flowing to the first cementation tank. The overflow from the surge tank will feed into a circuit consisting of two 10 ft x 17 ft cementation cone tanks, operating in series, where solution will contact shredded scrap steel. The copper in solution will react with the steel to produce “cement” copper, while the iron in the steel is dissolved. The cementation tanks will have a solution residence time of 15 minutes each. Precipitated copper “cement” slurry will be drawn periodically from the bottom of each cementation cone tank before being pumped to the double deck wash screen. Cement copper will be discharged, packaged, and sold as a final product.

Cementation tank overflow and cement copper wash water will be combined and most will be pumped to the dilute acid mix tank for use in the leach circuit. A portion will be bled off to maintain the water balance around the leach circuit. The bleed stream will be sent to an acid water treatment plant for neutralization. After a solid/liquid separation step, clear treated water will be recycled to the process water tank and sludge will be mixed with the leach residue (final tailings) for disposal.

17.4 Process Water

The water system for the process plant will consist of the following:

- Freshwater supply
- Firewater supply
- Seal water supply
- Process water supply

Freshwater will be delivered from the City of Mackay. There is an existing pipeline from City Spring to the City of Mackay Water Tank where it is proposed to tie in a project supply pipeline to convey fresh water to the proposed 357,000-gallon Fire/Fresh Water Tank. The conveyance from tie in point to the project tank

will require booster pump stations. This water will be used as make-up water for the process water supply, emergency firefighting supply, seal water, dust suppression, and water for the reagents make-up. The upper half of this tank will act as freshwater storage for the mill area, holding approximately 177,000 gallons. The lower half of the tank will be held in reserve for the fire suppression system, containing approximately 180,000 gallons to supply the fire suppression system at 1,500 gpm for two hours.

The processing facilities will draw seal water from the Seal Water Tank, which will be fed from the Fire/Fresh Water Tank. Seal water will be used by the slurry pumps in the plant.

The process water system will include a 24,000-gallon Process Water Tank. Process water will be supplied to the plant via three centrifugal pumps, two operating and one stand-by unit. This water supply will be used for the ball mill and cyclone feed pump box water requirements. The Vacuum Drum Filter concentrate, Cleaner Concentrate Thickener Overflow, Leach Feed Thickener Overflow, Neutralization Thickener, and the Fire/Fresh Water Tank all report to the process water tank.

18. PROJECT INFRASTRUCTURE

The Empire Project is identified as a 4,000 stpd copper plant as presented in this section. The major infrastructure elements crucial for operations, excluding those within the process plant, which are described in detail in Section 17.

18.1 Site Layout

The Empire Project is in the Alder Creek Mining District in Custer County, central Idaho. The property is approximately 3.3 miles southwest of Mackay, Idaho in the Big Lost River Valley between Arco, Idaho and Challis, Idaho. Figure 18-1 identifies the Empire mine's location.

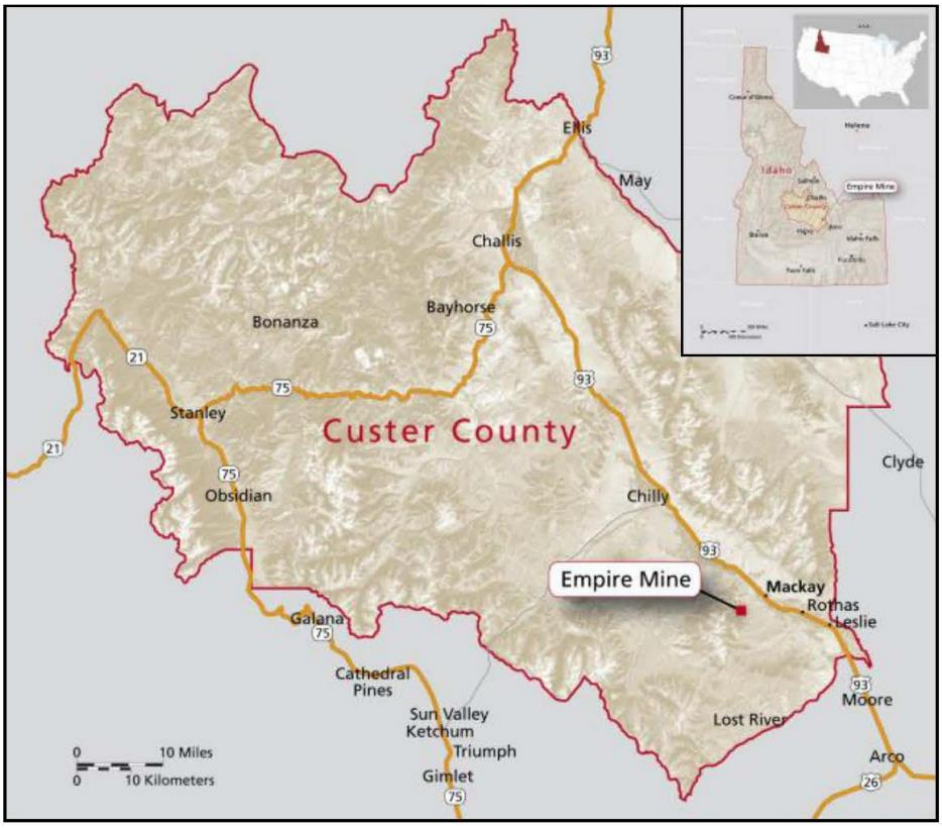


Figure 18-1: Empire Mine Project Location

Project infrastructure at Empire mine is designed to support the mining, processing, and dry stack tailings facilities. There are sufficient and appropriate areas within the site to accommodate mining facilities to include waste rock storage facility, processing facilities, and all applicable storage facilities. Infrastructure that is essential to the mining and metals production includes a crusher and conveyor circuit, stockpiles, access roads, haul roads, maintenance, and storage area, and supporting ancillary facilities. Figure 18-2 displays the overall site map that identifies major infrastructure.

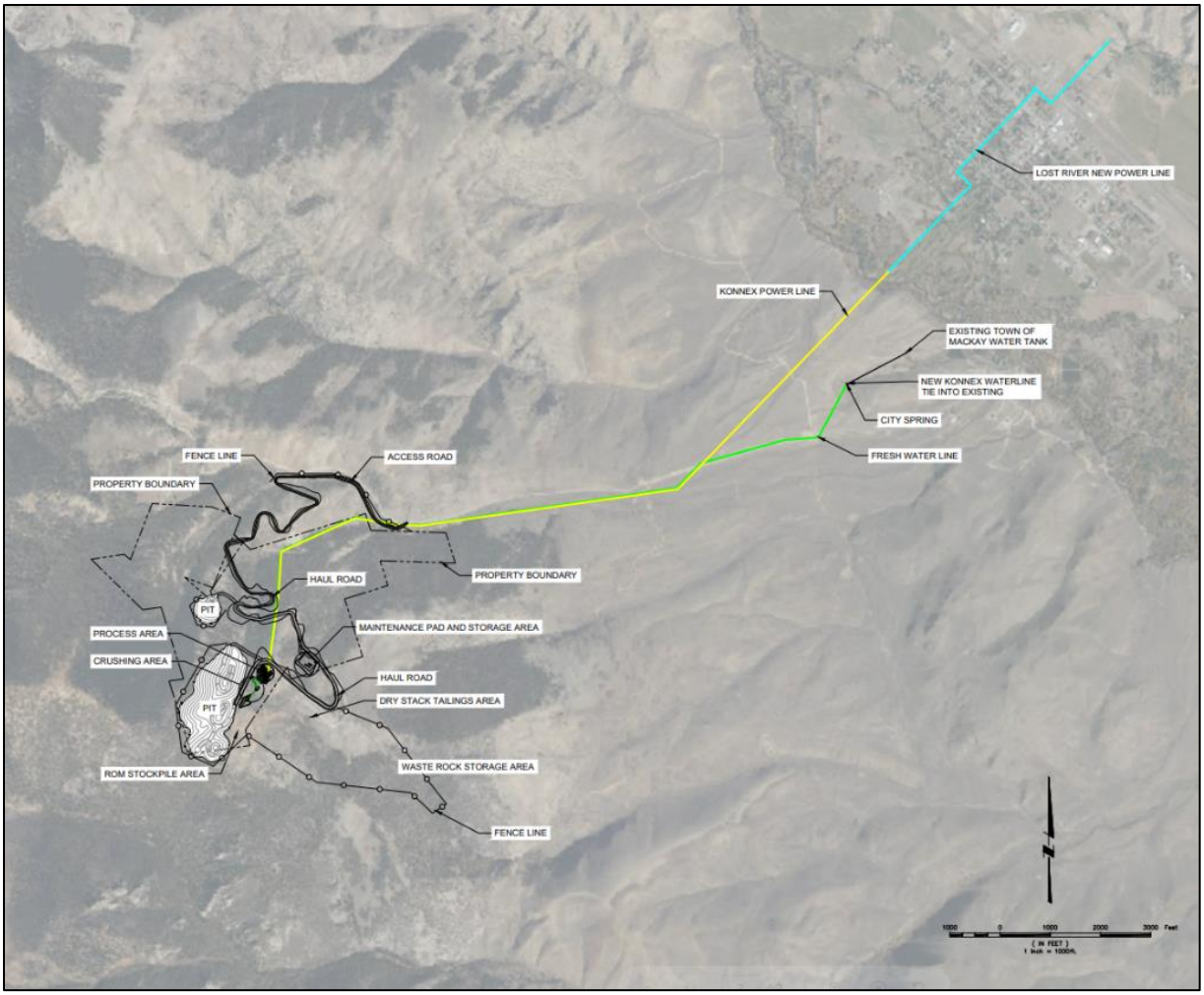


Figure 18-2: Site Layout Map

18.2 Area 2000 – Mine

18.2.1 Area 2300 – Mine Support Facilities

18.2.1.1 Waste Rock Storage Facility

The Waste Rock Storage Facility (WRSF) is shown above in Figure 18-2. The WRSF was designed with a 3:1 ultimate slope ratio, aiming to optimize layout and provide a starter embankment for the dry stack tailings facility. Design of the dry stack tailings facility is presented in Section 18.3. Lift heights for the WRSF have not been finalized, but a strategy for determining them will be developed in subsequent stages of the project. The WRSF will incorporate a single perforated drainpipe underdrain to intercept ground water flow and direct them to a discharge with a rip rap outlet at the toe to prevent erosion. Additionally, an underdrain measuring 16 ft wide by 3.5 ft deep, enveloped in geotextile to prevent clogging, will traverse the extent of the WRSF. The underdrain serves to facilitate drainage and mitigate saturation within the WRSF. Future design considerations for the WRSF will include slope stability analysis, compaction methods, and adherence to relevant environmental and regulatory standards. Table 18-1 outlines the design parameters of the WRSF and Figure 18-3 shows the layout of the WRSF.

Table 18-1: 4,000 stpd Waste Rock Storage Facility Parameters

WRSF Parameter	Value
Waste Rock Storage Capacity	10.5 MM tons
Average Waste Rock Bulk Density	125 lb/ft^3
Average Slope Ratio	3:1
Design Life	9 years

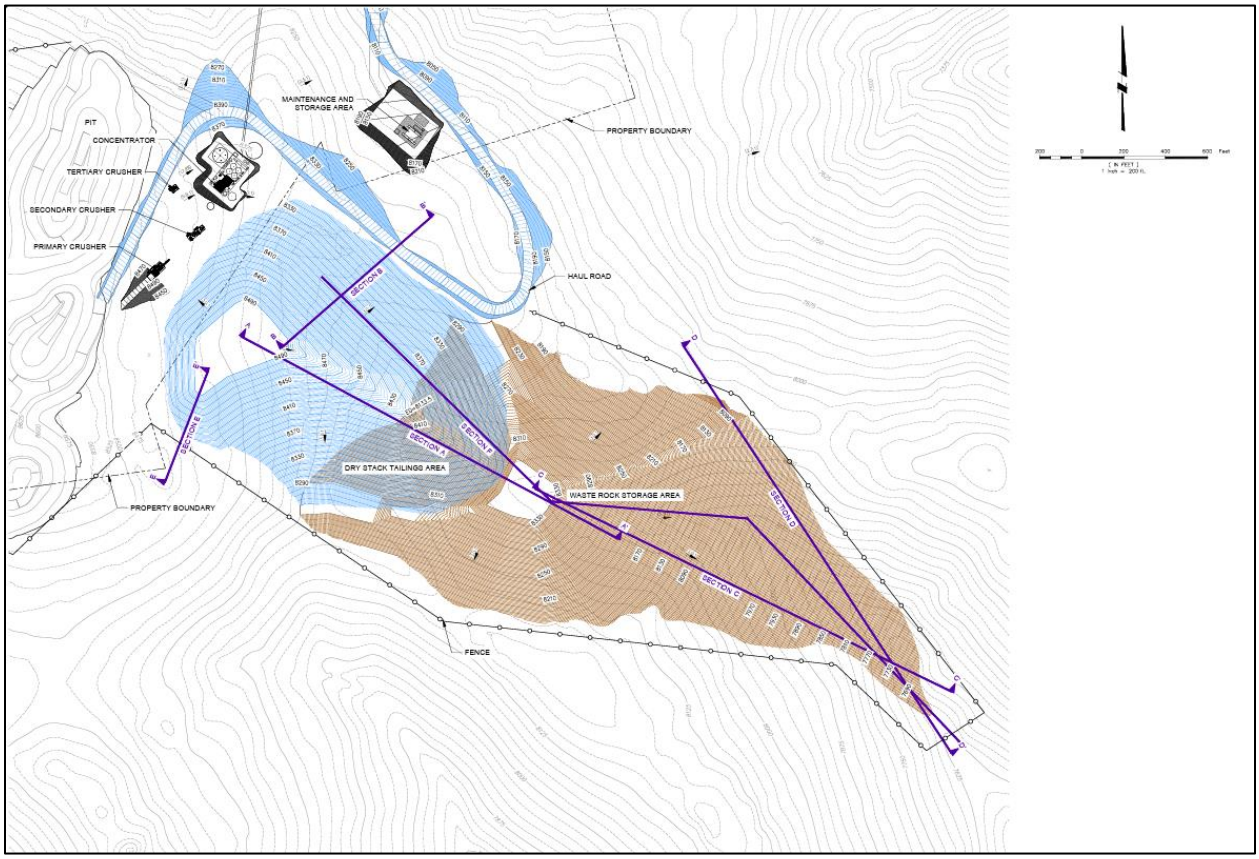


Figure 18-3: TSF and WRSF Layout

18.2.1.2 Truck Maintenance Pad

The truck maintenance pad, includes a concrete slab measuring 150 ft by 200 ft, will serve as the central hub for all essential maintenance operations. It will accommodate the truck shop, wash-down area, office space, and fueling station. See Figure 18-4 for general arrangement of the truck maintenance pad.

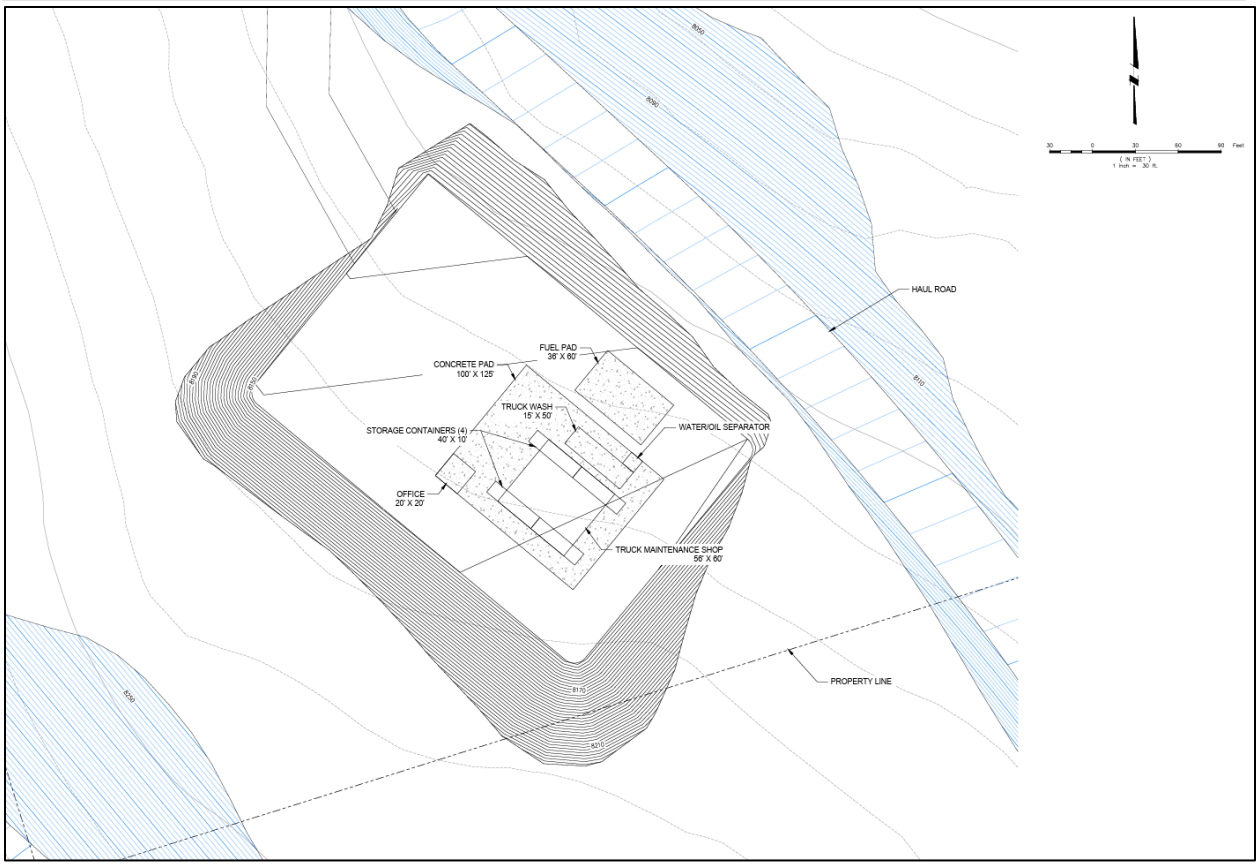


Figure 18-4: Truck Maintenance Pad General Arrangement

18.2.1.3 Truck Shop

The truck shop is a two-bay pre-engineered fabric building that will be placed on Sea Containers. The truck shop will house space for lubrication equipment, storage tankage for fresh and waste fluids, and a compressor area. The Sea Containers that the fabric truck shop rests on will be used for parts storage, hydraulic hose rebuild station and a tire changing station. This shop is intended for preventative maintenance and minor repairs. All major repairs will be completed offsite.

18.2.1.4 Office Trailer

The office trailer will be a 20 ft x 20 ft modular structure.

18.2.1.5 Truck Wash

The truck wash pad is an outdoor installation which will include spray monitors and wheel washes, a water heater, a sump for waste wash water, an oil-water separator, and a portable pressure washer.

18.2.1.6 Fuel Area

The fuel area will be located on the truck maintenance pad for easy access by articulated mine trucks, the lube truck, and other equipment using red dye diesel. Vehicles can simply pull onto the truck pad for refueling before exiting directly back to the haul road. To prioritize environmental safety, the fuel area will include a containment area to promptly catch any leaks or spills. It will be equipped with a dual-walled 15,000-gallon red dye diesel tank and a 1,000-gallon diesel exhaust fluid tank. See Section 18.4.7.2 for more discussion on the fuel area.

18.2.2 Area 2400 – Mine Support Services

Mackay offers standard municipal amenities including lodging and services, and a limited supply of food and hardware. The nearest major supply center is Idaho Falls, roughly 100 miles east of the Project area. Commercial air and rail service are both available in Idaho Falls, which is served by the Idaho Falls Regional Airport and the Eastern Idaho Railroad. Rail access is also available in the communities of Blackfoot and Pocatello, roughly 30 and 50 miles south of Idaho Falls, respectively. Unskilled and skilled labor can be found in Mackay, Idaho Falls, and a variety of other communities throughout the regional area.

18.3 Area 3500 – Cementation and Tails

Description of the cementation process is captured in section 17.3.5 of this report.

18.3.1 Area 3540 – Tailings Storage Facility

A total of 10.5 million tons of process tailings will be stored in the Dry Stack Tailings Storage Facility (TSF) over a design operating life of 9 years, with a nominal ore process rate of 4,000 stpd. The TSF, located southeast of the processing plant, will undergo staged raises using downstream tailings facility construction techniques, with the embankment constructed from the Waste Rock Storage Facility (WRSF) located further downhill. The WRSF will serve to buttress the TSF, ensuring its stability. The TSF can be seen in Figure 18-2.

Tailings from the series of leach tailings disc filters will undergo mixing with carbonate flotation concentrate, lime, and Portland cement in a PUG mill before being conveyed to the TSF via grasshopper conveyors to a radial stacker. The tailings will then be stacked on the TSF using a dozer. The TSF, a lined facility, will feature a pond at the bottom near the embankment. Operational and standby submersible pumps will be installed at the pond's bottom, reclaiming collected water back to the Process Water tank. Table 18-2 outlines the design parameters of the TSF and Figure 18-3 shows the layout of the TSF.

Table 18-2: 4,000 stpd Tailings Storage Facility Parameters

TSF Parameter	Value
Tailings Storage Capacity	10.5 MM Tons
Average Tailings Dry Density	98 lb/ft^3
Design Life	9 years

18.4 Area 4000 – Project Services

This section details supply and distribution of services and infrastructure outside of the processing plant. Figure 18-5 details the major utility layouts and infrastructure at the Empire Project.

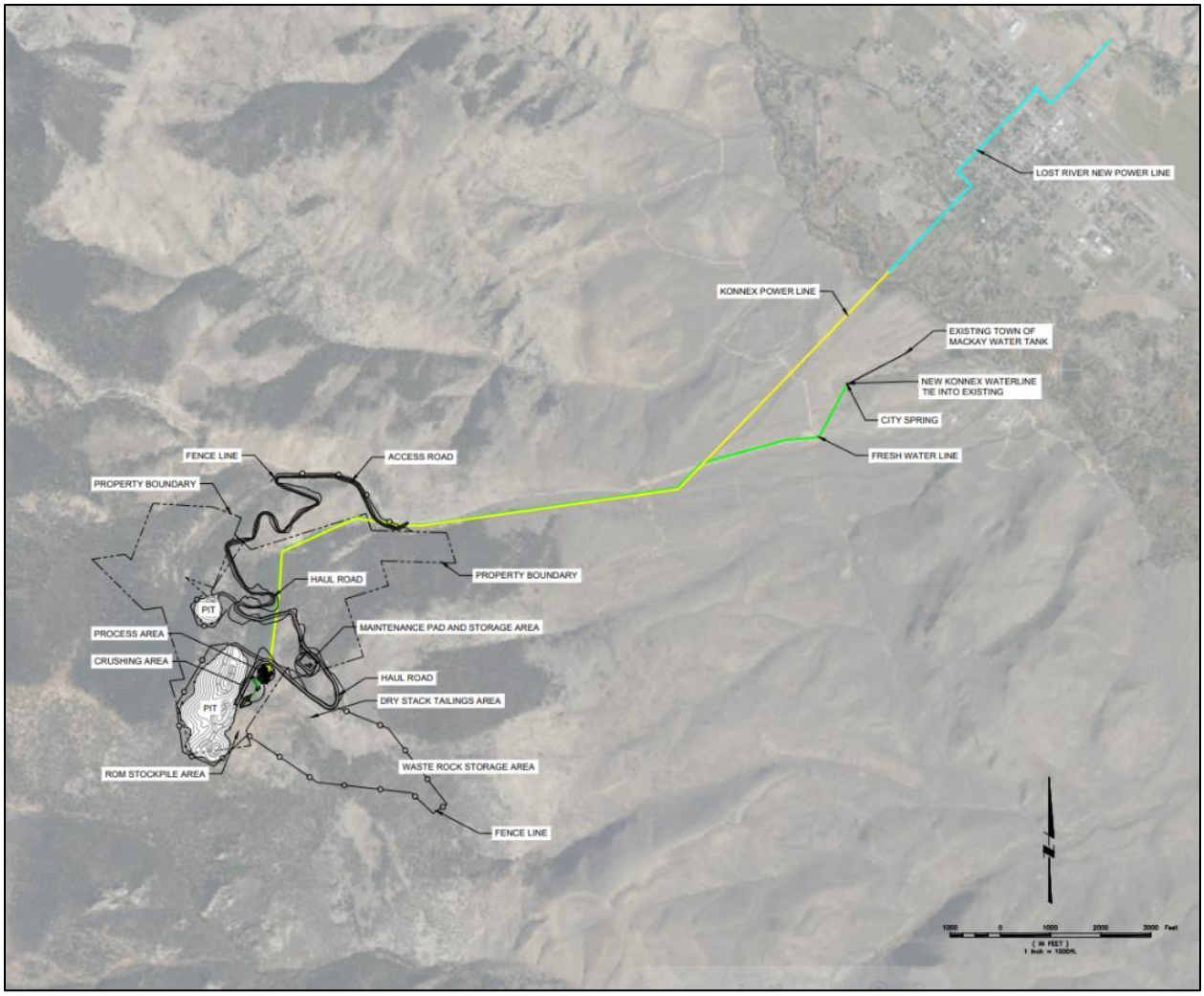


Figure 18-5: Utility Layout

18.4.1 Area 4100 - Power Supply

Power for the Empire mine will be provided by Lost River Electric Cooperative (LREC), located in Mackay. Several transmission lines run along the Big Lost River Valley from Arco and are tied into the main Idaho power grid, which is supplied by hydroelectric dams along the Snake River and is managed by Bonneville Power Authority. A substation located at Mackay taps both the 232 kV and 69 kV transmission lines. The connected load for the Empire mine is estimated to be 8 megawatts.

A new 24.9 kV power line will be installed from the Mackay substation to provide line power to the Empire mine site. An existing right-of-way and a 7.2 kV distribution line currently serves residential power on the west side of the valley. LREC will extend the new 24.9 kV line through the City of Mackay to the west side of the golf course.). Konnex will take ownership of the line at the west side of the golf course and supply power to the onsite substation.

The power line's route is planned to run almost due southwest from the west end of the golf course until it reaches the access road. From there, it will run adjacent to the access road before cutting across the Empire mine's property, following the most efficient path to the substation.

It is assumed that the current LREC substation located on the east side of Mackay is adequate and or upgrade cost incurred would be carried by LREC.

At the main Empire substation, power will be transformed to 4,160 V for distribution throughout the plant. Overhead power lines will supply medium voltage (4,160 V) and low voltage (480 V) to various process areas, catering to the needs of end-users. Electrical equipment rooms and motor control centers will be strategically located at the crushing plant, process plant, and truck maintenance area.

Step-down transformers will be installed to serve specific locations such as the truck shop, plant maintenance building, and guard house. Additionally, a 12 MVA transformer in the substation has been designed to meet the current demands of the Empire project while also accounting for future expected needs.

Pole mounted transformers will also be located along the overhead power alignment to provide power to fresh water pumps delivering freshwater from the City Springs to the Concentrator.

18.4.2 Area 4200 – Communications

The connection to telephone and internet services has not been confirmed at this time; however, telephone service is available in the City of Mackay. A Cellular Telemetry System will be used to communicate data exchange between the Process Plant and the City of Mackay. The system will incorporate a Master Telemetry Station, located in a switch room of the Process Plant, and a number of remote Telemetry Stations, located in remote equipment switchboards. The Master Telemetry Station will communicate with the Plant Process Control System via the preferred communications network and will communicate with the remote locations. Control of the remote equipment will be made by the Plant Process Control System, with sufficient data exchange to ensure correct operation of the equipment.

Fiber is an alternative that could be brought to site with relatively low cost by installing in parallel with the overhead power line servicing this project.

To ensure effective communication among personnel and equipment, a site-wide VHF radio network will be installed, equipped with multiple channels. Frequencies for this network will be assigned and approved by the Federal Communications Commission (FCC). This system will facilitate radio communication for both routine and emergency purposes, with mobile radios provided for operating and maintenance personnel to use outside office premises.

18.4.3 Area 4300 – Ancillary Facilities

Ancillary buildings necessary to support the Empire Project include an existing administration building (located in town), warehouse building, an assay laboratory, and the main gatehouse and truck scale. Other facilities include the truck shop for the mine fleet, a truck wash building, and a diesel storage and dispensing facility located outdoors.

18.4.3.1 Administration Building

The administration building is an existing single-story modular building located offsite in the City of Mackay. The administration building is approximately 60 ft by 40-ft (2,400 ft.2) and will house the administrative and supervisory personnel for the operation.

18.4.3.2 Main Assay Laboratory

The assay lab for the Empire mine will process daily production blasthole samples from the mine along with analytical data from samples in the processing plant. The lab building will be located either on site or adjacent to the site, at Konnex property located at or near City of Mackay.

18.4.3.3 Main Security Building & Truck Scale

Along the mine access road, a security building including motorized access gates will be installed. The security building will be a 12 ft x 12 ft modular structure. Person accessing the mine property will be required to pass through the security gate. The security building will also be used by security personnel for recording movement to and from the Site. The facility will include a motorized gate, pedestrian turnstile, and swipe card access.

Security personnel will control access to the mine and operate the truck scale. The scale will weigh all receipts of reagents and consumables as well as concentrate filter cake and cement copper production leaving the plant. See Section 18.4.6 for more information regarding site security.

18.4.4 Area 4400 – Access Road

The access to the Empire mine follows the existing Smelter Avenue from Mackay to the mine site. Road improvements along Smelter Ave from Mackay are not required until Konnex's property boundary. The mine access road, designed to be 25 ft wide, traverses across the mine property, where it transitions into a haul road leading to the truck maintenance area and processing facilities.

The mine access road and haul road has been designed for a maximum grade of 8% and will accommodate two-way traffic and oversize vehicles delivering mining equipment. Berms and drainage channels were incorporated into the design. Side slopes were designed at 2H:1V on fill sections and 1.5H:1V on cut sections. Culverts were also installed beneath the road where drainages cross to prevent washouts.

18.4.5 Area 4500 – Water Management

Stormwater run-on will be diverted away from disturbed areas of the Project. Current layout at or near the top of the ridge will require smaller diversions and ponds to adequately handle stormwater events. Contact stormwater will be collected in ponds, which may be used for makeup water in the processing facilities. The roject water balance will be prepared in the next level of study to design the pond volumes. It is anticipated that culverts will be required on the access and haul roads where drainages cross to prevent washouts.

18.4.6 Area 4600 – Security

Section 18.4.3.3 details the security building and main access gate.

Persons accessing the mine property will be required to pass through the security gate. The general site fencing will be standard five-strand barbed wire range fencing or jack leg fencing. The fence will keep ATV traffic, and livestock out of the mine site.

Security will be monitored with cameras from the guardhouse and any other cameras associated with buildings. Anomalies picked up by the cameras will be investigated by a site security guard who will also be responsible for the safety of the mine.

18.4.7 Area 4700 – Site Utilities

18.4.7.1 Area 4710 – Water

18.4.7.1.1 Water Treatment Plant (WTP)

The Water Treatment Plant will receive both cementation overflow and copper wash from the cementation process, with only a portion of each being necessary, at a nominal flow rate of 650 gallons per minute (gpm).Test work is ongoing for further design will to be completed in the Feasibility Study.

The primary function of the water treatment plant will be to neutralize the acidic effluent from cementation. The basic unit operations in such a neutralization plant would consist of a lime silo, possibly a slaker, one or more neutralization reactor tanks and a clarifier. The clarifier would use gravity settling to separate a clear, neutralized overflow for recycle and a sludge for disposal. The water treatment plant could be largely automated and monitored by personnel in the concentrator. Treated water will be reused in the process.

18.4.7.1.2 Freshwater

The freshwater requirement for the Konnex Empire Mine Project will be approximately 200 gpm. Fresh water will be purchased from the City of Mackay. Fresh water is sourced from a proposed pipe tie in point of existing pipe line from the City Spring to an existing city water tank. The water will be pumped and conveyed through a carbon steel pipeline, following the path of the powerline up to the processing plant. A Pump station will be necessary at the tie in point near the city water tank and one more along the route to the processing plant, with the potential for additional stations depending on the final design.

A 6-in carbon steel pipeline will be installed, following the route parallel to the power line and access road for approximately 17,600 ft up to the mill site. This pipeline will serve to convey freshwater to the Fire/Fresh Water tank positioned at the mill location. Along the pipeline route, multiple booster stations will be integrated to facilitate the efficient transportation of fresh water to the mill.

18.4.7.1.3 Potable Water

Potable Water will not be required for the Konnex Empire Mine Project. Portable toilets will be brought to site to provide facilities for employees. Eyewash and safety showers will be self contained units that can be refilled with bottled water from the City of Mackay. Bottled water will be provided onsite for employees.

18.4.7.1.4 Fire Water

There is an existing pipeline from City Spring to the City of Mackay Water Tank where it is proposed to tie in a project supply pipeline to convey fresh water to the proposed 357,000-gallon Fire/Fresh Water Tank. The conveyance from tie in point to the project tank will require booster pump stations. This water will be used as make-up water for the process water supply, emergency firefighting supply, seal water, dust suppression, and water for the reagents make-up. The upper half of this tank will act as freshwater storage for the mill area, holding approximately 177,000 gallons. The lower half of the tank will be held in reserve for the fire suppression system, containing approximately 180,000 gallons to supply the fire suppression system at 1,500 gpm for two hours. The fire water system will consist of a jockey pump, diesel pump, and electric pump. Fire water will be distributed to the mill through a fire sprinkler system. Additionally, strategically positioned fire hydrants near the mill will ensure easy access for local fire trucks.

18.4.7.2 Area 4720 – Fuel Supply

The fuel storage and management system will be located on the truck maintenance pad for easy access by articulated mine trucks, the lube truck, and other equipment using red dye diesel. Vehicles can simply pull onto the truck pad for refueling before exiting directly back to the haul road. To prioritize environmental safety, the fuel area will include a containment area to promptly catch any leaks or spills. The fuel storage and dispensing area is equipped with all necessary equipment for the safe transfer of fuel and additives. Each tank will be connected to a fuel distribution system to include a filtration system and fuel dispensing equipment. Off-road red dyed diesel will be stored in one 15,000-gallon dual-walled tank. Diesel exhaust fluid (DEF) for use in all diesel-powered equipment and vehicles will be stored in one 1,000-gallon diesel tank. The fuel storage and dispensing area is equipped with all necessary equipment for the safe transfer of fuel and additives.

Fuel and diesel exhaust fluid will be delivered by truck from a local Mackay company.

18.4.8 Area 4800 – Waste Disposal

Portable toilets will be used on site to accommodate the employees. Cleaning services will be sourced from a local company, which will also manage sewage disposal by transporting it to the sewage treatment facility in Mackay. For waste management, dumpster and roll-off bins will be utilized for garbage storage. These containers will be supplied by a local company, responsible for both their provision and the hauling of garbage to a nearby facility as required. The routine generation of solid and hazardous waste, inherent in mining and milling activities, will be meticulously managed in compliance with local and state regulations.

Any hazardous waste generated at site would be placed in drums, on pallets, labelled, and stored on the truck maintenance pad or in the mill building in a designated location. The pallets would be placed in an area offering secondary containment where the material would be stored until such time as it could be hauled offsite by a licensed contractor for disposal in a safe and environmentally sound manner.

18.4.9 Area 4900 - Plant Mobile Equipment

The plant mobile equipment to be used for the project include:

Table 18-3: Plant Mobile Equipment

Description	Total Units
Telehandler	1
Front End Loader - 4.25 cy bucket	1
Pickup Trucks (0.5 Ton)[1]	6
Pickup Trucks (0.75 Ton) – Diesel[1]	1
Crew Vans	2
Crane - 28 ton	1
Skid Steer	1
Forklift	1
Dozer	1
Scissor Lift, rough terrain	1
Compactor Sheepsfoot	1
Subtotal	**17**

Notes:

1. *Konnex has already purchased light duty pickup trucks.*

18.5 Area 5000 – Project Infrastructure Development

18.5.1 Area 5100 – Site Preparation

The following site preparations are included for the development of the Empire Project surface infrastructure works area:

- Disturbed areas include:
  - Concentrator and mill
  - Crusher area
  - Truck Maintenance Pad
  - Access road and haul roads.
  - Powerline & freshwater line footprint
  - Open Pits
- Clearing and grubbing for disturbed areas as required; soil will be removed and stockpiled for use during site reclamation.
- Cut and fill to prepare for disturbed areas: cut material will be reused for fill materials wherever possible. Bulk earthworks are designed to minimize the import of fill materials
- Site grading and road water management.
- Installation of powerline and site water supply.
- Installation of chain link, barbed wire fences, and jack leg fences on the site.
- Access gates will be installed at the site entrance.

18.5.2 Area 5600 – Bulk Transport

Bulk transport in and out of site will be weighed on a truck scale near the security building. The truck scale will be located on a dedicated off take from the main access road. The site weighbridge will be capable of weighing a maximum truck or tanker of 100 short tons.

19. MARKET STUDIES AND CONTRACTS

19.1 Markets

The operation will produce marketable grade copper concentrate containing precious metals (gold and silver) and cemented copper.

The amount of products produced will vary from year to year. The ongoing proposed test work will determine yearly production of the two saleable products.

Copper is readily saleable but long-term contracts are beneficial.

19.2 Contracts

No contracts are in place as of the completion of the PFS. However, it is recommended that discussions with smelters and/or traders be initiated prior to the completion of the FS and/or start of construction.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The anticipated environmental permits and social impacts of this project are summarized based on the conceptual Project design presented in this PFS and current regulatory requirements. The conceptual approach to closure and reclamation and the associated estimated costs are based on the anticipated development for the Project. This information will require review and reassessment if changes to the scope, area, or design of the Project occur as Project planning and design advance.

20.1 Environmental Studies

The Project is located on private land controlled by Konnex with ancillary facilities on public land managed by the US Forest Service (USFS) and US Bureau of Land Management (BLM). The mine and mill will be located on private property. The access road, waste rock dump and tailings storage facility will be located on public land previously affected by historical mining within the Salmon-Challis National Forest (SCNF). Utilities, including an overhead powerline and buried water line, will cross public land managed by BLM. The water line is planned within an existing road corridor.

As required by Idaho Rules Governing Mined Land Reclamation, a Project Plan, consisting of a Reclamation Plan and Operations Plan, will be submitted to the Idaho Department of Lands (IDL) for review and approval under the Idaho Joint Review Process (IJRP). Approval of construction on federal public lands will be subject to environmental review under the National Environmental Policy Act (NEPA).

To support Project planning for construction, operations, closure, and reclamation; permitting; and the environmental impact analysis required for the NEPA review, a comprehensive program of studies characterizing the existing natural resources, cultural resources, and socioeconomic resources in the Project area will be submitted to the agencies for review.

Historical mining activity, livestock grazing, USFS road development, timber harvest, recreation, and wildfires have affected natural resources in the Project area. The Empire Mine is considered a brownfield development of a historic mine. Thus, while the term “baseline” is used in this section, it is noted that for some resources, a more appropriate description is “pre-Project” characterization, because pre-disturbance baseline conditions can no longer be directly assessed in all areas.

20.1.1 DEQ Preliminary Assessment

A preliminary assessment and site inspection (PA/SI) of the Empire Mine by the Idaho Department of Environmental Quality (DEQ) identified environmental concerns at the historic mill site, which occupies approximately 6.5 acres near the northeast corner of the patented claim block (DEQ, 2005, 2017). Potential hazardous waste violations at the mill were related to unlabeled containers, oil and diesel fuel spills, and leakage from lead-acid batteries. Arsenic concentrations in a concentrate pile and mill tailings could pose risks to recreational visitors by incidental ingestion, inhalation of particulates, and dermal contact.

20.1.2 Environmental Baseline Studies

Environmental baseline studies that have been completed, are in progress/ongoing, or are pending to support Project permitting and EA preparation include the following:

- Air Quality and Meteorology (ongoing)
- Cultural Resources (pending for off-patent disturbance)

- Geochemistry (ongoing)
- Geological Resources
- Geotechnical Hazards (ongoing)
- Ground Water Hydrology (ongoing)
- Noise
- Visual Resources
- Soil Resources and Reclamation Cover Materials
- Surface Water Hydrology (ongoing)
- Vegetation, Botanical Resources and Non-Native Plants
- Wildlife Resources and Wildlife Habitat
- Access and Transportation (in progress)

20.2 Permitting

Permitting requirements for this Project are based on the Project description presented in this PFS and include the following features to be constructed on patented claims, except as noted:

- Two mine pits
- Mine roads and haul roads
- Explosives storage magazines
- Tailings storage facility that will encroach on the Salmon-Challis National Forest (SCNF)
- Waste rock storage facility that will encroach on the SCNF
- Mine access road to encroach on public land managed by the SCNF
- Electrical substation and overhead distribution lines that will cross public land managed by the Bureau of Land Management (BLM) and the SCNF
- Water transmission line on public land managed by BLM and the SCNF
- Modular crushing plant
- Ball mill, flotation mill and cementation plant
- Process water distribution, storage and treatment facilities
- Truck maintenance pad
- Fuel storage and distribution facilities

- Stormwater control facilities on private and public land

20.2.1 Idaho Joint Review Process

The IDL is responsible for implementation of the IJRP, which was established in 1996 to coordinate and facilitate mine permitting between all permitting agencies (federal, state and local). The IJRP was developed by IDL through Memorandums of Understanding (MOUs) between IDL and other agencies, with the intent to have all the permitting agencies review and approve the same documents at or near the same time. The IJRP enables pre-analysis coordination in interagency review of a plan submittals including alternatives, draft and final permits, and financial assurance coordination.

The IJRP promotes concurrent reviews to reduce permitting timelines. The IJRP may play a key role in achieving two primary permitting goals: (1) increased communication and cooperation between the various involved governmental agencies, and (2) reduced conflict, delay, and costs in the permitting process. The table below lists the permits that are anticipated will be required for this project.

Table 20-1: List of Environmental Permits, Reviews and Approvals

Lead Agency	Permits, Reviews and Approvals	Project Application	Project Phase
Federal			
BLM	Right of Way Grant	Access road	Preconstruction
	Environmental Assessment	Access road	Preconstruction
USFS	Road Use Permit	Forest System Roads	Preconstruction
	Special Use Permit	Facilities on public land	Preconstruction
	Environmental Assessment	Facilities On Public Land	Construction
USEPA	Spill Prevention Plan (SPCC)	Mine Fuel Facility	Preconstruction
	EPA Waste Generator ID	Mill	Operation
	SARA Title III – EPCRA	Mill	Operation
	TSCA – TRI	All Areas	Operation
State of Idaho			
IDL	Operations Plan	All Areas	Preconstruction
	Reclamation Plan	All Areas	Preconstruction
	Financial Assurance	All Areas	Construction
DEQ	Air Quality Permit to Construct	Mill	Operation
	Stormwater Construction General Permit Notice and SWPPP	All Areas	Preconstruction
	Stormwater Industrial Multi-Sector General Permit Notice and SWPPP	All Areas	Operation
	Groundwater Point of Compliance Determination	Mine	Construction
IDWR	Water Right	Mill	Preconstruction
SHPO	Cultural (SHPO) Clearance	Public Land Disturbance	Preconstruction
Local			
Bldg Dept	Building Permits	Mill	Construction

20.2.2 State of Idaho Permits

The Project Plan will be submitted to IDL. The Project Plan will describe the proposed methods of construction, operation, closure, and reclamation. Key components of the Project Plan will be the descriptions of mining and processing, water and waste rock management plans, and the closure and reclamation plan. The Project Plan will also describe the best management practices and environmental design features for protection of the environment.

IDL will coordinate with federal and other state agencies under the IJRP to review the Project Plan for conformance with regulatory requirements. Federal land management agencies will conduct an environmental review as described in Section 20.2.3. State agencies will review the plan and coordinate with Konnex to issue permits that will address air quality, water quality, and water usage.

20.2.3 Federal Environmental Review

Several Project components will be located, at least in part, on public lands managed by the US Forest Service (USFS) and US Bureau of Land Management (BLM). The major Federal laws applicable to Project permitting are the General Mining Law of 1872, as amended (30 USC §§ 22 et seq.), the Organic Administration Act of 1897, as amended (16 USC §§ 473-475, 477-482 and 551), and the Federal Land Policy and Management Act of 1976 (FLPMA), as amended (43 USC §§ 1701 et seq.). The USFS implementing regulations for these laws, 36 Code of Federal Regulations (CFR) 228 Subpart A, and the BLM minerals surface management regulations, 43 CFR 3809, require that locatable mineral prospecting, exploration, development, mining and processing operations, and associated means of access, be conducted in a manner that minimizes adverse environmental effects on federal surface resources. The USFS and BLM must conduct an analysis of environmental effects in accordance with the National Environmental Policy Act of 1969, as amended (NEPA; 42 United States Code [USC] §§ 4321 et seq.). The NEPA analysis must consider all relevant environmental statutes, including, but not limited to, the Federal Clean Air Act of 1970, as amended (42 USC §§ 7401 et seq.), the Clean Water Act of 1972, as amended (33 USC §§ 1251-1388), the Endangered Species Act of 1973, as amended (16 USC §§ 1531-1544), the Wilderness Act of 1964, as amended (16 USC §§1131 – 1136), the Wild and Scenic Rivers Act of 1968, as amended (16 USC §§ 1271-1287), and the National Historic Preservation Act of 1966, as amended (54 USC 300101 et seq.).

The USFS and BLM will jointly prepare an Environmental Assessment (EA) of activities proposed on public land described in the Project Plan, as required by the NEPA regulations promulgated by the federal Council on Environmental Quality (CEQ) (40 CFR Parts 1500–1508). The major phases of the NEPA review process are scoping (inviting review and comment on the Plan by the public and other interested parties to define the scope of issues to be addressed), preparation of a draft EA and public comment period, and issuance of a Final EA, Finding of No Significant Impact (FONSI) and Decision Notice/Record. If one or more significant issues arise that will require additional analysis under an Environmental Impact Statement (EIS), the EA may not be issued as the EIS process proceeds.

After the Project Plan is submitted to the USFS and BLM, the agencies will review the Plan to identify preliminary issues. The agencies will publish a 30-day public notice in the local newspaper and direct invitations to cooperating agencies (Federal and State of Idaho) Tribes, and persons or parties to provide comments for consideration during EA development. Information collected during formal consultation and scoping will be used to identify and prioritize the resources and issues to be considered and to develop alternatives to be used in the EA analyses.

Data from baseline studies and other available reports relevant to the Project area will be used to develop a draft EA. The public and other parties will be notified of availability of the EA, draft FONSI and draft Decision Notice/Record for comment on USFS and/or BLM websites. The comment/objection period will be 45 days long, but agencies can grant requests to extend the comment/objection period by 30 days. After the comment period, the agencies will attempt to mediate objections before issuing the Decision Notice/Record.

20.3 Geochemical Characterization

An environmental geochemistry baseline study was conducted to characterize the chemical behavior of mined rock during weathering and predict leachate water quality from the mine and waste rock dump (Valley Science and Engineering, 2021). Samples were selected from Konnex's exploration database. Static testing was completed on 251 samples selected with the highest sulfur content (greater than 400 ppm), which represented 18% of the 1,374 available samples. Static testing included acid-base accounting (ABA), net acid generation testing (NAGT), and Meteoric Water Mobility Tests (MWMTs). Tests were completed on coarse reject samples from the assay lab.

20.3.1 Potential Acid Generation

ABA consists of a series of tests to evaluate the balance between potentially acid-generating (PAG) minerals and acid-neutralizing minerals in a rock or soil sample. Based on the ABA tests, samples were classified as PAG using two different criteria: the net neutralization potential (NNP) and the neutralization potential ratio (NPR). Using these criteria, six samples are PAG based on NNP and 14 samples are PAG based on NPR (including the 6 classified as PAG based on NNP). Samples classified as PAG consisted of endoskarn, exoskarn, and barren late stage instrusive dike (dike) rock types.

Table 20-2: Net Neutralization Potentials by Rock Type (n-251)

Rock Type	Quantity	PAG	Uncertain	Non-Acid Generating
Endoskarn	70	1	7	62
Exoskarn	55	3	19	33
Granite Porphyry	44	0	11	33
Limestone	38	0	0	38
Dike	30	2	26	2
FeOX Breccia	11	0	11	0
Granite	2	0	1	1
Quaternary Alluvium	1	0	0	1
Total	251	6	75	170
% Total	100%	2%	30%	68%

The net acid generation test (NAGT) was performed on 81 samples classified from the ABA test as PAG or Uncertain to independently evaluate ARD potential. The NAGT produced acidic leachate from 15 samples, including 7 samples from the ABA procedure.

Overall, 22 samples were classified as PAG from the ABA and NAGT testing. Based upon the mine block model, 11 PAG samples are ore grade (including 4 in the pit and 7 outside the pit) and 11 PAG samples are waste rock (with 5 in the pit and 6 outside the pit).

20.3.2 Leaching Tests

Leaching tests by the MWMT procedure were performed on five samples classified as PAG, including 4 ore samples and 1 waste sample, of which 2 samples (one ore sample and one waste sample) fall within the pit shell. Leachate from one of the ore (granite porphyry) samples from outside the pit met all of the Idaho groundwater standards. Leachate from another ore (exoskarn) sample, from inside the pit, exhibited acidic pH and elevated copper, iron, manganese, sulfate and total dissolved solids (TDS). Leachate from a waste rock (dike) sample from inside the pit and an ore sample from outside the pit (iron oxide breccia) were nearly neutral pH but had elevated mercury concentrations (0.0162 mg/L and 0.0424 mg/L, respectively, compared to groundwater criterion of 0.002 mg/L). Leachate from one of the ore samples (endoskarn) exhibited slightly alkaline pH and elevated concentrations of manganese, sulfate and TDS.

Kinetic humidity cell tests (HCTs) were performed to evaluate leachate water quality changes due to weathering of the rock samples. HCTs were conducted between 22 to 41 weeks on five samples, including 4 ore samples and 1 waste rock sample of which 3 ore samples were previously tested by the MWMT procedure. At 22 weeks, testing was terminated on 2 samples that produced very dilute, non-acidic leachate (one of the ore samples from outside the pit and the waste rock sample from inside the pit). Mercury was detected in leachate from the waste rock sample from the first week of HCT (0.0028 mg/L, compared to groundwater standard of 0.0020 mg/L) and also in the MWMT (0.0162 mg/L). The second ore sample (from inside the pit) produced acidic leachate with elevated copper, manganese and TDS from the MWMT and HCT with the HCT leachate also containing elevated aluminum and cadmium. The third ore sample (from outside the pit) produced alkaline leachate with elevated manganese, sulfate and TDS in the first week of HCT and in the MWMT with manganese persisting in HCT testing for 22 weeks. The fourth ore sample (from inside the pit) produced leachate with nearly neutral pH with elevated copper and selenium for the first few weeks and persistent cadmium, manganese and zinc for 41 weeks.

20.3.3 Waste Rock

The geochemical study concluded that less than two percent of the drill samples, all containing sulfide minerals, would be classified as PAG. Sulfide-enriched rock would be processed in the flotation mill instead of disposed as waste rock. Based on these results, the ARD risk for the Project is negligible.

Leaching tests of rock samples classified as PAG by MWMT and HCT procedures, including 4 ore samples and 1 waste rock sample, indicates that samples containing ore-grade copper and zinc sulfide minerals can produce alkaline or acidic drainage with elevated salinity, copper, and zinc. Both leachate tests of the waste rock sample classified as dike material contained elevated mercury, but only in the initial leachate from the HCT procedure. The dike lithotype is only 11 cubic yards and 0.0001% of the deposit that will be mined, so the slightly elevated concentrations of mercury detected in leachate from dike samples is not significant.

20.3.4 Tailings

One representative tailings sample from the locked cycle testing was submitted by Forte Analytical to ACZ Laboratory on March 21, 2024 for extraction by the synthetic precipitation leaching procedure (SPLP). The tailings sample was analyzed for percent solids while the extracted leachate was analyzed for pH, temperature, conductivity, total dissolved solids (TDS), total cyanide, common anions in natural water (alkalinity, chloride, fluoride, nitrate+nitrite, sulfate), 4 metals that are common cations in natural water (calcium, magnesium, potassium, sodium), silica, and 21 trace metals. The results are exhibited below with Idaho Groundwater Standards.

Table 20-3: Analyses of Tailings Leaching Test by Synthetic Precipitation Leaching Procedure

Analyte	Units	Value	Idaho Groundwater Standard
pH	std units	**4.9**	6.5-8.5
Alkalinity	mg/L	<2	
Chloride	mg/L	<1	250
Conductivity	umhos/cm	2190	
Cyanide, total	mg/L	<0.101	0.2
Fluoride	mg/L	1.22	2
Nitrate/nitrite-N	mg/L	0.026	10
TDS	mg/L	2280	500
Sulfate	mg/L	1310	250
Aluminum	mg/L	<0.01	0.2
Antimony	mg/L	0.00166	0.006
Arsenic	mg/L	0.00455	0.05
Barium	mg/L	0.03	2
Beryllium	mg/L	<0.0004	0.004
Boron	mg/L	<0.005	
Cadmium	mg/L	0.0111	0.005
Calcium	mg/L	602	
Chromium	mg/L	<0.001	0.1
Copper	mg/L	2.52	1.3
Iron	mg/L	0.121	0.3
Lead	mg/L	0.00435	0.015
Magnesium	mg/L	1.15	
Manganese	mg/L	0.364	0.05
Mercury	mg/L	0.00026	0.002
Molybdenum	mg/L	0.0245	
Nickel	mg/L	0.00645	0.1
Potassium	mg/L	1.72	
Selenium	mg/L	0.001	0.05
Silica	mg/L	38.7	
Silver	mg/L	<0.0002	0.05
Sodium	mg/L	1.94	
Thallium	mg/L	0.00063	0.002
Uranium	mg/L	<0.0002	0.03
Vanadium	mg/L	<0.001	0.002
Zinc	mg/L	0.836	5

The leachate extracted from the tailings sample was acidic (pH 4.9), moderately saline (TDS of 2280 mg/L), and containing mainly calcium sulfate (1912 mg/L representing 84% of the TDS). Since the leachate had higher concentrations than Idaho Groundwater Standards for TDS, sulfate, copper and manganese, and a lower pH than the standard range, the potential influence of these parameters on groundwater quality will need to be evaluated during permitting. The Idaho Groundwater Quality Rule contains provisions for any mine operator to request a point(s) of compliance, which could allow the standards to be exceeded beneath the mining area if the mine is operated accordingly while in compliance with applicable permits.

20.4 Water Management

Based on the current hydrological understanding, pit dewatering is not anticipated to be needed. Exploration drill holes have not encountered groundwater. Existing underground mine workings do not have any observable mine water drainage, nor do the historical records indicate dewatering activity.

20.5 Social and Community Impact

The following summary of potential social and community impacts of the project were adapted from a socioeconomic report prepared for the project in 2021 (SWCA Environmental Consultants, 2021). The impacts were scaled down from the previous larger scale mine plan.

20.5.1 Cultural, Social, and Economic Resources

Custer County spans 4,937 square miles and is a rural mountain county with a total population of approximately 4,169; the population decreased by approximately 2.5% between 2010 and 2019, and the county currently has a population density of 0.8 person per square mile. The population of Custer County has fluctuated throughout the twentieth century. Approximately 91% of the population in Custer County predominantly identify as White Alone. Approximately 14% of the families in Custer County live below the poverty line, and approximately 18% of the people in the county live below the poverty line.

The largest employment sector in Custer County is government (16%), followed by farm employment (13%) and hospitality services (13%). The basic industries in Custer County are farming; forestry, fishing, and related activities; mining, quarrying, and oil and gas extraction; utilities; information; arts, entertainment, and recreation; accommodation and food services; other services; and government.

Between 2000 and 2020, the unemployment rate in Custer County peaked in 2013 (9.3%) while peaking in the reference area (7.7%) and Idaho (9.0%) in 2010. Between 2010 and 2019, the median household income for Custer County decreased by approximately 2.5% while the median household income in the reference area and Idaho increased by approximately 7.8% and 20.2%, respectively.

The average property tax rates in Custer County in 2020 were 0.730% for urban districts and 0.497% for rural districts. Property tax collections in Custer County totaled $2,865,595 in 2010 and $4,106,834 in 2019. Travel and convention (lodging) taxable sales in Custer County totaled $5,111,541 in 2010 and $9,577,680 in 2019. The State of Idaho applies a tax for the privilege of mining or receiving royalties from mining operations in Idaho. The mine license tax rate is 1% of the value of ores mined or extracted and royalties received from mining. Idaho collected $2,723,273 in mine license taxes in 2010 and $116,862 in 2020.

In 2019, there were 3,277 total housing units in Custer County, with 54.1% of these units occupied. The rental vacancy rate in 2019 was 17.1%. Approximately 75% of the housing units in Custer County in 2019 were classified as single unit, detached. The second-most common housing unit type in the county in 2019 was mobile homes. The median monthly rental rate in Custer County in 2019 was $608. In 2020, Custer County had approximately 378 hotel and motel rooms as well as approximately 245 recreational vehicle sites.

There are two school districts in Custer County: Challis Joint School District #181 and Mackay Joint School District #182. The schools in these districts are Challis Elementary School, Stanley Elementary/Junior High School, Challis Junior/Senior High School, Mackay Junior–Senior High School, and Mackay Elementary School. Other public facilities and services in Custer County include fire protection, law enforcement, health care facilities, recreational facilities, and wastewater treatment.

20.5.2 Employment

Because mining is a base industry in Custer County, it is anticipated that each new job could create between 2.5 to 3.4 new support jobs in the county. Under a 126-job scenario for the Empire Project, there would be between 315 and 428 new support jobs. If all the support jobs involved people relocating to Custer County—which is unlikely—the total employment would represent an approximately 11% to 14% increase in the county's total population. This would also represent a 0.20% to 0.25% increase in the reference area's total population.

20.5.3 Taxes and Royalties

Project spending for the Empire Project would include initial capital costs, sustaining capital costs, mine operating costs, and processing plant operating costs. A portion of this spending would occur in Custer County and the reference area, which would result in economic benefits to the local economy. In addition to direct spending, taxes will be paid to Custer County, the State of Idaho and the federal government. Please see Sections 21 and 22 for further details on tax estimates for the project.

20.5.4 Community Impacts

The number of available housing units, hotel/motel rooms, and RV sites in Custer County would likely be able to accommodate the Empire Project's 126 to 157 new mining employees. However, because of the rural nature of Custer County, employees would be more likely to find available housing units in the more populated areas of the reference area outside of Custer County.

The relatively small increase in population from this project would likely have a minor impact on public safety in the county, as a small increase in population could result in a small increase in public safety incidents, such as potential traffic accidents associated with mine-related traffic or potential accidents at the mine site. Similarly, the relatively small increase in population would likely have a minor impact on the availability of health care and human services in the county, as a small increase in population would likely result in a small increase in the need for health care and human services. The relatively small increase in population would also likely have a minor impact on municipal services and government facilities. The public water system, wastewater treatment facilities, waste collection, and utilities services in Custer County are unlikely to be strained by the potential 3% to 4% increase in population from new mine employees and proposed mine operations.

20.5.5 Community Engagement and Relations

Konnex frequently engages with local economic development groups, Lost Rivers Economic Development (LRED) and Butte County Economic Development (BCED) in Custer and Butte Counties and are committed to finding further practical ways to support their activities. They partner with businesses and government while taking into account community interests on behalf of local residents.

Konnex held a public information meeting on December 1, 2021. In February 2022, the Konnex Community Advisory Team, KCAT, was formed to provide a liaison between the company and residents of Custer and Butte Counties. KCAT meets quarterly and includes 3 company representatives and 10 local citizens representing local businesses, municipalities, and public land management. KCAT provides a forum for stakeholders to promote mutual benefits of the Empire project to the community and identify/discuss community concerns. KCAT also has a student scholarship committee.

The recreational experiences currently provided in the vicinity of the proposed mine would likely continue. Konnex is designing the project to accommodate other local activities such as tourism and community

development. Because of the long history of mining in the area, there is also an opportunity for Konnex to work with federal and local agencies/recreation groups to provide for interpretive signage or recreational improvements that describe past mining history and proposed mining operations and further promote recreation and tourism within the study area.

20.5.6 Tribal Consultation

The NEPA review will also include Government-to-Government consultation between USFS and the Shoshone-Bannock Tribe. During this consultation, a determination will be made if traditional cultural properties, cultural landscapes, sacred sites, or tribal resource collection areas exist in the areas that would be impacted by the Project.

20.6 Closure and Reclamation Planning

The Project closure and reclamation plan will address all disturbances and infrastructure associated with the Project. The primary goals of the closure and reclamation plan will include the following:

- Reclaim disturbed and affected land to its original or another beneficial use.
- Re-shape disturbed areas and re-establish diverse and self-sustaining plant communities.
- Minimize erosion.
- Remove hazards.
- Isolate, remove or control toxic materials.
- Control water runoff and maintain water quality.
- Rehabilitate aquatic and terrestrial habitats.

This closure and reclamation plan is preliminary, and the level of detail is consistent with a PFS level of design. The final Project Plan will include the closure and reclamation plan as a key component. The closure and reclamation standards and requirements, conceptual approach based on the current plan for Project development, cost estimate and financial assurance requirements are discussed in the following subsections.

20.6.1 Reclamation Standards and Approach

Prior to commencing mining operations, the closure and reclamation plan must be approved by USFS and IDL. The closure and reclamation standards and requirements set forth in 36 CFR 228 Subpart A, the Idaho Rules Governing Mined Land Reclamation (IDAPA 20.03.02), as well as best management practices developed by IDL for protection of water quality will be incorporated into the closure and reclamation plan.

During the closure and reclamation phases, site activities including water management and treatment will continue to be conducted to maintain compliance with Project permits. Monitoring and maintenance during these phases will address water quality monitoring, site stabilization, revegetation, weed control, and public access controls for health and safety.

Key assumptions used to develop the conceptual approach for closure and reclamation included the following:

- The mine pits (located on private land) will not be backfilled so that all the waste rock will be placed in a single external dump.
- The mine pit high walls will be stable, and not require regrading.
- The lower half of the waste dump will be constructed with maximum slope of 3 Horizontal to 1 Vertical (3H:1V), whereas the upper half of the waste dump will require regrading from 1H:1V to 3H:1V.
- The waste rock dump would be regraded to a maximum 3H:1V slope, covered with 12 inches of growth media and revegetated.
- Dry tailings will be placed in the storage facility adjacent to the wasterock dump and graded during operations to a maximum slope of 3H:1V to mitigate erosion. Concurrent reclamation of the tailings will be applied to the maximum extent practical.
- Water management in the tailings storage area would include collection and treatment of leachate and any other contact water during operations.
- A low permeability cover will constructed on the tailings storage area to minimize infiltration of precipitation and production of leachate. The cover will be 60-mil geomembrane overlain by 12-inches of drain rock and 18 inches of growth media.
- Long-term monitoring and maintenance for the Project would involve monitoring for water quality as required by applicable regulations, monitoring for successful establishment of vegetation and appropriate stormwater management, and control of invasive plant species.

20.6.2 Reclamation and Financial Assurance Cost Estimates

Key assumptions used to develop the closure cost estimate include the following:

- Closure requirements will be comparable to other recently permitted projects in the region.
- Costs assume the work will be completed by mining company personnel and equipment rather than by an independent contractor.
- An alternative cost estimate was generated for financial assurance (FA) for the work to be performed by a 3rd party retained by the government using rented equipment, based on prevailing commercial costs and government administrative oversight.
- Costs include the applicable staffing, operations, and maintenance costs during closure, mobilization costs, engineering and design, and contingencies.
- The estimated cost includes a budget for post-closure activities.

The total capital cost estimate for reclamation was estimated at $1.3 million USD and a Reclamation Reduction of $3.3 million. As presented in the financial model discussed in Section 22.

Prior to commencing mining operations, financial assurance (FA) for closure and reclamation costs must be provided to IDL (IDAPA 20.03.120) and USFS (36 CFR 228.13). The FA is generally developed to account for both the short-term and long-term closure and reclamation phases of the Project. The short-term phase typically addresses the activities immediately following mine closure, including removal of infrastructure no longer required for post closure management of the site and initial reclamation of disturbed areas. The alternative cost estimate for FA was $7.2 million USD. Assuming a bond rate of 3 percent, the bonding is estimated at $216,000.

The long-term phase includes long-term activities such as monitoring, maintenance, and water treatment and is typically developed using a Net Present Value (NPV) determination. The NPV determination allows the amount of a FA to be discounted for long-term obligations generally exceeding a five-year period.

21. CAPITAL AND OPERATING COSTS

The Empire Project is identified as a 4,000 stpd copper flotation and cementation plant as presented in this section. For the purposes of understanding how the mine will operate, Table 21-1 details the Project based on the principal operating time periods.

Table 21-1: Operating Periods

Principal Assumption	Unit	Parameter
Construction Period	Years	1.25
Commissioning & Ramp Up	Months	2
Mine Life	Years	8
Closure Period	Years	1
Operating Days	Days / Year	350

Estimated capital and operating costs are summarized in this section and are prepared by Forte Dynamics engineers and consultants as follows:

- Open Pit Mining: Hard Rock Consulting (HRC)
- Process Plant: Forte Dynamics (Forte)
- Tailings Dam: Forte Dynamics
- Infrastructure: Forte Dynamics and Hard Rock Consulting
- Fresh Water: Forte Dynamics
- Power: Forte Dynamics
- Reclamation: Valley Engineering (VSE)
- Owners Costs: Forte Dynamics, Konnex, and Valley Engineering

Costs are presented in Q1 of 2024 in US dollars.

21.1 Capital Costs Summary

LoM capital requirements are estimated at $83.8 million, and an initial Capital of $62.6 million as summarized in Table 21-2. Initial capital and working capital for items purchased in the first two years before production and is estimated to be $63.5 million. The methodology for the development of the capital cost estimates was primarily performed by obtaining new equipment cost and factoring using published values to account for total install cost. The full equipment list for both mining and process was analysed by Konnex. Konnex identified available used equipment on the market, and reduced cost for both mobile and mechanical equipment that reduced LOM CAPEX by approximately $16 million. Some of the equipment including the mill has already been purchased. All install cost remained based on factors from new equipment cost. CAPEX methodology is described in more detail in the following sections.

Table 21-2: Estimated Capital Cost Summary (USD $000s)

Capital Costs ($000s)	Initial	Sustaining	Total LOM
2000 Mine Infrastructure and Equipment	$6,611	$16,409	$23,020
3100 Crushing and Conveying	$4,327	$0	$4,327
3200-3400, 3520 & 3530 Process Plant	$22,766	$0	$22,766
4000 Infrastructure and Utlilites	$9,728	$0	$9,728
3540 Tailings and Ponds	$9,742	$6,474	$16,216
Total Direct Costs	**$53,173**	**$22,883**	**$76,056**
8000 Owner Costs, EPCM, Commissioning	$6,760	$0	$6,760
9100 - 9400 First Fills, Spares	$2,666	$0	$2,666
8500 Reclamation & Bonding	$0	$1,300	$1,300
9700 Contingency	$0	$0	$0
10000 Salvage Value	$0	($2,903)	($2,903)
Total Indirect Costs	**$9,427**	**($1,603)**	**$7,824**
TOTAL Capital	**$62,600**	**$21,280**	**$83,880**
9500 Working Capital	$891	($891)	$0
TOTAL CAPITAL/Working Capital	**$63,491**	**$20,389**	**$83,880**

21.2 Capital Cost Estimating Methodology for Mining

The mine capital costs for this study are estimated from HRC's in-house database of projects and studies, Cost Mine Service and budgetary quotes for mining equipment. The mine production fleet was quoted as new equipment from Caterpillar equipment dealer. Costs for major equipment rebuilds on engines, transmissions and final drives have been included as sustaining capital. A mine shop area has been planned with a fabric building for the initial years and a separate wash down area. A fuel bay system has been included within the plant infrastructure costs. Replacement units are added as they reach the end of the useful life hours of the machine.

21.3 Mine Capital Costs

Table 21-3 lists the detailed costs for the initial, sustaining and total mine capital. At the end of the mine life the mine production mobile equipment is assumed to be sold with a salvage value of 10% of the new costs.

Table 21-3: Mine Capital Cost Summary (USD $000s)

Description	# Initial Units	# Additional Units	# Replacement Units	# Total Units	$/Unit	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
982 Loader	1	1	0	2	$358	$358	$358	$716
Production Drill	1	0	0	1	$462	$462	$0	$462
PreShear Drill	1	0	0	1	$284	$284	$0	$284
Haul Truck - 745	3	3	0	6	$350	$1,050	$1,050	$2,100
16' Grader	1	0	0	1	$400	$400	$0	$400
Water Truck	1	0	0	1	$320	$320	$0	$320
448hp Dozer	1	0	0	1	$420	$420	$0	$420
347hp Dozer	1	0	0	1	$312	$312	$0	$312
Lube/Fuel Trk	1	0	1	2	$175	$175	$175	$350
Service Truck	1	0	1	2	$125	$125	$125	$250
SkidSteer	1	0	1	2	$60	$60	$60	$120
Pickups mine G&A	2	0	0	2	$40	$80	$0	$80
Pickups Drilling	1	0	0	1	$35	$35	$0	$35
Pickups Blasting	1	0	0	1	$35	$35	$0	$35
Pickups Maint	2	0	1	3	$55	$110	$55	$165
Pickups Engineering	2	0	0	2	$35	$70	$0	$70
Pickups Geology	1	0	0	1	$35	$35	$0	$35
Light Plants	6	0	1	7	$15	$90	$15	$105
Gen set Genset-1	1	0	0	1	$30	$30	$0	$30
Compressor	1	0	1	2	$17	$17	$17	$34
Dump Truck	1	0	0	1	$50	$50	$0	$50
Excavotor	1	0	0	1	$320	$320	$0	$320
Dewatering Pump	1	0	0	1	$26	$26	$0	$26
Welders	2	0	1	3	$5	$10	$5	$15
Total Equipment						**$4,874**	**$1,860**	**$6,734**
Mine Equipment Rebuilds						$0	$14,549	$14,549
Mine Truck Shop	1	0	0	1	$1,736	$1,736	$0	$1,736
Total Before Contingency						$6,611	$16,409	$23,020
Contingency						$0	$0	$0
Total Mine Capital after Contingency						**$6,611**	**$16,409**	**$23,020**

21.4 Capital Cost Estimating Methodology for Process and Infrastructure

A PFS design has been developed for a 4,000 stpd plant as the basis for the capital cost estimate. The methods used to estimate capital in the capital cost estimate are summarized in the following sections.

The capital cost estimate for the Process Plant and Ancillary Facilities is shown in the table below, Table 21-4.

Table 21-4: Capital Cost Estimating Methodology – Process Plant and Ancillary Facilities

Bulk Commodity	Capital Methodology
Mechanical Equipment	A detailed mechanical equipment list, with supply pricing based on budget quotations, public sources, and internal body of knowledge. New cost used for factoring to account for install. The equipment costs ultimately were based on Konnex findings of available used equipment, some of which such as the Mill is already purchased.
Concrete	MTOs based on 3D model and unit rates for Maintenance area only. All other process areas were factored from the supplied new mechanical equipment supply price for all other areas
Structural Steel	Cost factored from new equipment
Platework	Percentage factor of the supplied new mechanical equipment supply price for certain areas
Tankage	Included on mechanical equipment list on preliminary design calculations with budget quotations.
Piping	Percentage factor of the supplied new mechanical equipment supply price, internal body of knowledge for certain areas
Electrical	Supply and installation pricing based on budget quotations for substation. Percentage factor of the supplied mechanical equipment supply price for certain areas
Instrumentation & Control	Percentage factor of the supplied new mechanical equipment supply price

Subsequently estimate factors, by area, were back calculated for each bulk commodity as a percentage of the mechanical equipment supply cost estimate. In turn, the resultant estimate factors were critiqued against published data and industry experience.

21.4.1 Mechanical Equipment Total Installed Capital Cost Estimating Methodology

The basis for estimating the mechanical equipment supply costs was largely based on budgetary pricing from vendors. The vendors were provided with preliminary specifications and/or data sheets for mechanical equipment items. The budget quotations received from vendors are expected to have an accuracy equal to ±25%. All other minor equipment items were priced from Forte's database of costs from recent similar sized projects or from Cost Mine Service, along with other public sources. The new equipment cost was utilized for factor basing of installation costs. Konnex identified an opportunity to reduce cost by identifying available used equipment currently available. Some equipment, including the Mill has already been purchased. The total CAPEX presented utilizes the used cost found for mechanical equipment and does not include the purchased Mill.

A construction factor was incorporated into the cost estimation for every piece of mechanical equipment within the Empire mine design. These considerations were necessary to address items not by definition included in the prefeasibility level design. The construction cost factors account for various line items such as instrumentation, building structures, concrete foundation, installation, and related expenses, ensuring a comprehensive assessment of construction costs.

The methods used to estimate capital in the cost estimate are summarized in the following sections. The cost estimate for the mechanical equipment features the methodology shown in the table below. Forte leveraged the Canadian Institute of Mining and Metallurgy's, "Mineral Processing Equipment Costs and Preliminary Capital Cost Estimation Book" (Mular, 1978) and industry experience to estimate that the total installed cost of the equipment is 2.5 times the equipment cost, while the construction cost of the equipment would be 1.5 times the original cost of the equipment. Table 21-5 includes the items that are included in the cost factor that were not included in the PFS design.

Table 21-5: Construction Cost Basis for the Processing Plant

Plant Component	Construction Cost Relative to New Equipment
Equipment	1.00
Equipment Installation	0.22
Process Piping	0.05
Electrical	0.20
Instrumentation	0.06
Process Buildings	0.37
Aux. Buildings	0.07
Plant Services	0.05
Site Improvements	0.02
Field Indirects	0.13
Project Management	0.26
Total Installed Equipment Costs	2.43
Total Installed Equipment Cost Used in Study	2.5
Total Construction Cost Used in Study	1.5

21.4.2 Pre-Configured Mechanical Equipment Total Installed Capital Cost Estimating Methodology

The construction capital cost estimate for plug-and-play equipment or outdoor tanks employs a distinct basis for estimating construction capital costs compared to standard mechanical equipment on the Empire Project. This construction factor applies specifically to outdoor tanks and silos, conveyors, as well as modular buildings/trailers, as outlined in Table 21-6.

Table 21-6: Construction Cost Basis Other Areas of the Capital Cost Estimate

Equipment	Construction Cost Relative to Equipment
Outside Tanks and Silos	1.0
Conveyors (Mobile & Stationary)	1.0
Security/Office Trailer	0.5

21.4.3 Freight Costs

Freight was not accounted for as a direct line item in the capital cost estimate; however, it is understood that the construction cost estimate factors will capture the freight costs.

21.4.4 Other Areas Capital Cost Estimate

Table 21-7 shows the capital cost estimating methodology for areas outside of mechanical equipment within the process facilities and ancillary facilities.

Table 21-7: Cost Methodology for Other Areas of the Capital Cost Estimate

Area/Sub-Area	Cost Methodology
Facility 2000 - Mine	
Area 2300 – Mine Support Facilities	Drawings developed for the WRSF, maintenance pad and priced largely on a per unit basis
Facility 3000 – Process Plant	
Area 3540 – Tailings Storage Facility	Drawings developed for the TSF and priced largely on a per square foot basis, budgetary quote for lining material
Area 3610 – Water Treatment	Engineering Estimate without design
Facility 4000 – Project Services & Infrastructure	
Area 4100 – Power Supply	Based on length of overhead power lines, load list and budgetary quotes, and comparable projects.
Area 4200 – Communications	Price not included; telemetry cost included in mechanical equipment cost factors
Area 4400 – Access Road	Based on MTOs and unit costs from comparable projects
Area 4500 – Water Management	Costs included in road unit cost; pond cost (if necessary) will be estimated in the next level of study. Water management in pit included in OPEX, as no permanent infrastructure required.
Area 4600 – Security	Drawings developed for fencing, priced on a per foot basis.
Facility 8000 – Management, Engineering, EPCM Services	
Area 8100 – EPCM Services	
Area 8100 – Feasibility Study	The FS Cost is not included in the Capital Estimate
Area 8100 – Management	0.5% of total cost of project excluding Pre-Production Costs, EPCM, and mobile equipment
Area 8100 – Engineering	4.0% of total cost of project excluding Pre-Production Costs, EPCM, and mobile equipment
Area 8100 – Trade-offs/Value Engineering	0.5% of total cost of project excluding Pre-Production Costs, EPCM, and mobile equipment
Area 8100 – Project Services	0.5% of total cost of project excluding Pre-Production Costs, EPCM, and mobile equipment
Area 8100 – Construction Management	4.0% of total cost of project excluding Pre-Production Costs, EPCM, and mobile equipment
Area 8100 – EPCM Fee	1.0% of total cost of project excluding Pre-Production Costs, EPCM, and mobile equipment
EPCM Site Establishment	0.2% of total cost of project excluding Pre-Production Costs, EPCM, and mobile equipment
Area 8200 – External Consultants/Testing	Cost Provided by Owner
Area 8300 – Commissioning	Commissioning costs based on 3% of the total mechanical equipment cost 3%
Area 8400 – Owner's Team	Cost Provided by Owner
Area 8500 – Environmental/Legal	Cost Provided by Owner
Area 8600 – Project Insurances	Capital not included in estimate. Insurances already held by Owner
Facility 9000 – Preproduction Costs	
Area 9100 – Preproduction Labor	0.25% of the total direct costs of the project
Area 9200 – Commissioning Expenses	0.5% of the total direct costs of the project
Area 9300 – Capital Spares	5.5% of the total mechanical equipment costs of the project
Area 9400 – First Fills	Estimated from budgetary quotes and internal body of knowledge with detailed storage areas and consumptions.
Area 9500 – Working Capital and Finance	Excluded from this study
Area 9600 – Escalation and Foreign Currency Exchange	Excluded from this study
Area 9700 – Contingency Provision	Contingency is not included
Area 9800 – Management Reserve Provisions	Excluded from this study
Facility 10000 – Asset Sale	
Area 10000 – Asset Sale	10% of mechanical equipment, plant mobile equipment, and electrical equipment

21.4.5 Quantity Development and Unit Rates

The basis for the development of supply and installation costs of bulk commodities is discussed in the following section. Bulk commodities include civil, concrete, structural steel, etc. which will be used in the construction of the Empire mine. These costs were largely derived based on an estimate of material take-off (MTO) quantities which were multiplied by a unit rate for each type of material. The unit rates were calculated using Forte's standard methods including obtaining current market rates from contractors, and historical data.

21.4.5.1 Civil

Preliminary bulk earthwork quantities were estimated using civil 3D modelling software. The 3D modelling accurately calculates earthworks volumes utilizing the existing topography and proposed design levels. Structural excavation and backfill required for concrete structures are included in the concrete quantities. The TSF was cost utilizing an area unit cost for the footprint. All quantities were categorized by standard type of work classification. Unit rates for this work classification were developed from the in-house rates database. This rates database is constantly maintained so as to be current and has proven to be sufficiently accurate over several recent projects. The availability of water and local earthworks materials was taken into account in to the development of unit rates.

21.4.5.2 Concrete

Concrete cost were primarily factored from new equipment costs. In the case of the truck maintenance slab quantities were calculated from footprint drawings and an assumed thickness. Concrete quantities were categorized by standard classes of concrete including slabs on grade and concrete with containment joints. Unit pricing was obtained from industry sources by standard classification, each having an assessment of formwork, props, bracing reinforcing, embedment's, joints in slabs plus a miscellaneous allowance for curing, formwork hardware and other sundries. Concrete supply was costed at a rate deemed to include plant control testing, some admixtures and out of hours pouring. All other concrete costing in process area was included in the factor of mechanical equipment per Table 21-6.

21.4.5.3 Structural Steel

The estimate for the supply and installation of structural for project was factored based on a percentage of the total new mechanical equipment supply and installation cost.

21.4.5.4 Tankage

Budgetary quotes were received for major tanks, with vendors being supplied preliminary specifications and/or data sheets. Tanks that did not receive bids or were added/modified after the bidding process underwent costing based on a unit rate for steel square footage derived from the provided quotes.

21.4.5.5 Piping

The estimate for the supply and installation of process piping was factored based on a percentage of the supplied new mechanical equipment price. Piping supply and construction costs were also developed for the fresh water pipeline and the TSF return pipeline from similar project.

21.4.5.6 Electrical

The estimate for the supply and installation of electrical components was factored based on a percentage of the total new mechanical equipment supply and installation cost. Budgetary quotes were received for the substation located at the mill, with vendors being supplied preliminary specifications and/or data sheets. A cost basis of a comparable project for unit cost of overhead power was applied to alignment of proposed power distribution from the Mackay Golf Course to the Mill. Per conversation with Lost River Electric Company (LREC) the overhead powerline from LREC substation to the Golf Course tie in point would be completed by LREC.

21.4.5.7 Instrument & Control

The estimate for the supply and installation of instrumentation and controls was factored based on a percentage of the supplied new mechanical equipment price.

21.4.6 Area 8000 - Management, Engineering, EPCM Services

Facility 8000 will cover the indirect costs associated with the management of the project from detailed design through to handover to operations. Included within this section will be the EPCM team, external consultants, commissioning team, owner's team and any costs for licenses, fees, legal costs and insurances.

21.4.6.1 Area 8100 – EPCM Services

This area includes the costs of engaging the services of one or more contractors to perform the engineering, procurement and construction management for the project. The cost estimation does not include the costs for performing a Feasibility Study.,.

21.4.6.2 Area 8200 – External Consultants/Testing

This area is a cost that allows for the engagement of any environmental, accounting, Human Resources/Industry Relations or Health, Safety, Environment and Community (HSEC) consultants that might be required through the execution phase of the project.

21.4.6.3 Area 8300 – Commissioning

Area 8300 is concerned with the costs for the management and engineering associated with commissioning and was derived, for the Process Plant, as 3% of the total new mechanical equipment supply costs.

21.4.6.4 8400 – Owner's Team

Area 8400 contains costs associated with the owner's team located either on site and or in the project office. Owner's Team cost include but are not limited to sanitation facilities, road signage, training programs, and communications.

21.4.6.5 8500 – Environmental/Legal

Area 8500 will capture the costs of costs of licenses, fees and legal costs that would need to be expended throughout the execution phase of the project. This includes permitting costs expected to get the project operational.

21.4.6.6 8600 – Project Insurance

No costs have been included for Project Insurances. Konnex currently holds insurances; however, the amount of funds to be included in this area will be dependent on Konnex's criteria for an acceptable risk profile and, as such, is subject to interpretation by Konnex.

21.4.7 9000 Preproduction Costs

Facility 9000 will cover the indirect costs associated with direct labor during commissioning, the purchase of spare equipment and replacement of equipment damaged during commissioning. Areas 9500 to 9800 are sums of money associated with working capital, corporate reserves, escalation and exchange rate fluctuation, contingency and management reserve.

21.4.7.1 Area 9100 – Preproduction Labor

Preproduction labor covers the costs that are not part of Construction Contracts, not part of Commissioning, not part of post-handover operating ramp up costs but are for costs that may arise prior to operations taking over the Project in an operating context. This area is proposed for minor plant modifications and additions deemed necessary to achieve Project handover status.

21.4.7.2 Area 9200 – Commissioning Expenses

Commissioning expenses are intended to cover the power, materials, labor and spare parts that are associated with making plant modifications, additions and operations during the commissioning period.

21.4.7.3 Area 9300 – Capital Spares

Capital spares are all spares which are non-consumables. These are large items that are not expected to be used; however, these items must be kept in spares for the project due to long lead times, high cost and process importance. These items include, but are not limited to, spare motors, mill and crusher liners, screens, conveyor drives, pulleys, and belts.

21.4.7.4 Area 9400 – First Fills

First fills accounts for the first fill of the primary reagents within the processing facilities, including but not limited to lime, cement, sulphuric acid, flotation reagents, grinding media, and fuel.

21.4.7.5 Area 9500 – Working Capital and Finance

Working Capital will be an allowance for a sum of money to be left for use after the plant is operational before the revenue stream is stable. Project working capital provides for estimated normal timing delays associated with receipts and disbursements of cash, with such amounts being fully recovered by the end of the project life.

21.4.7.6 Area 9600 – Escalation and Foreign Currency Exchange

Escalation and Foreign Currency Exchange allowances will be necessary to cover potential inflation and fluctuation of foreign currencies from the date of this study until actual transaction dates. Such allowances have not been included in the estimate as provision for this will be made by Konnex.

21.4.7.7 Area 9700 – Contingency Provision

Capital cost contingencies were not included for the purposes of this report.

21.4.7.8 Area 9800 – Management Reserve Provision

The management reserve provision is a measure of the accuracy of this cost estimate and is a portion of additional money that would not be available to the project manager but will be held in reserve by Konnex to cover unforeseeable and uncontrollable events including, but not limited to strikes, unusual weather conditions, premium payments arising from accelerated construction programs to recover lost time. A reserve for such potential costs has not been included in the estimate as provision for this will be made by Konnex.

21.4.8 10000 Asset Sale

The Asset Sale encompasses an estimated salvage value of 10% for all mechanical equipment, plant mobile equipment, and electrical equipment. This salvage value represents the projected residual worth of these assets at the end of their useful life.

21.5 Process Plant & Infrastructure Capital Costs

21.5.1 3000 Process Plant

The estimated process plant and tailings storage facility capital costs are shown in Table 21-8. Initial capital totalling $36.1 million is estimated for the process plant and initial phase of tailings storage facility; with a total capital cost including sustaining capital for tailings storage facility of $42.6 million required.

Table 21-8: Estimated Process Plant Capital Cost Summary (USD $000s)

Area	Total Costs	Equip. Costs	Install Costs
3100 - Crushing	4,327	2,252	2,075
3200 - Grinding	6,842	528	6,313
3300 - Flotation	7,266	3,233	4,033
3400 - Leach	2,496	1,216	1,280
3500 - Cementation & Tails	12,463	2,988	9,476
3600 - Reagents	2,697	643	2,054
Sustaining Capital - TSF	6,474	570	5,904
Total	**42,565**	**11,430**	**31,135**

21.5.2 4000 Project Services & Infrastructure – General

Project services and infrastructure capital costs are shown in Table 21-9. Project services and infrastructure are estimated to have a LoM capital cost of $11.7 million.

Table 21-9: Estimated Project Services & Infrastructure Capital Cost Summary (USD $000s)

Area	Total Costs	Equip. Costs	Install Costs
4000 - Project Services	11,772	5,310	6,462

21.5.3 8000 Management, Engineering, EPCM Services

Management, engineering, and EPCM Services are shown in Table 21-10. Management, engineering, and EPCM Services are estimated to have a LoM capital value of $6.8 million.

Table 21-10: Estimated Management, Engineering, and EPCM Services Capital Cost Summary (USD $000s)

Area	Total Costs
8100 - EPCM	5,973
8200 - External Consultants/Testing	0
8300 - Commissioning	601
8400 - Owners Team	186
8500 - Environment/Legal	0
8600 - Project Insurances	0
Total	**6,760**

21.5.4 9000 Pre-Production Process Plant Costs

Pre-Production Costs are shown in Table 21-11. Pre-Production Costs are estimated to have a LoM capital value of $2.7 million.

Table 21-11: Pre-Production Costs Capital Cost Summary (USD $000s)

Area	Total Costs
9100 - Preproduction Labor	218
9200 - Commissioning Expenses	437
9300 - Capital Spares	1,102
9400 - First Fills	910
9500 - Working Capital and Finance	0
9600 - Escalation and Foreign Currency	0
9700 - Contingency Provision	0
9800 - Management Reserve Provisions	0
Total	**2,666**

21.6 Operating Costs

The operating costs include the ongoing cost of operations related to mining, processing, tailings disposal and general administration activities. The operating costs were determined based on Fortes's and HRC's in-house database of projects and studies and budgetary quotes or estimates for consumables and supplies. Following the AISC guidelines, LOM average Cash Operating Cost is projected to be $2.27/lb of CuEq sold. The AISC LOM average base case Total Operating Cost (including royalties) is expected to be $2.44/lb CuEq. The total AISC summary per ton of mill feed and per pound of CuEq sold is expected to be $33.35/t and $3.15/lb respectively as shown in Table 21-12.

Table 21-12: Total Operating Cost Summary (USD)

Operating Costs	$/CuEq lb	$/ton Ore	$/ton Mined	$/ton Moved
Total Mining	$0.76	$8.05	$2.99	$2.69
Total Processing	$1.30	$13.78		
Total Site General & Administration	$0.18	$1.87		
Mine License & Property Tax	$0.03	$0.30		
Cash Operating Costs	**$2.27**	**$24.00**		
Royalties	$0.17	$1.81		
Total Cash Costs	**$2.44**	**$25.81**		
Capital	$0.71	$7.54		
All-in Sustaining Costs	**$3.15**	**$33.35**		

21.6.1 Mine Operating Costs

Mine operating costs are calculated in detail by equipment, consumables, supplies, services and manpower requirements based on the mine schedule. Equipment costs are calculated based on required hours of operation to meet the production schedule and hourly costs for equipment components, supplies, consumables and manpower. The operating costs are calculated monthly in the mine schedule throughout the mine life, so a detailed analysis of the equipment requirements can be calculated. Diesel costs were estimated at $3.00/gallon. Mine maintenance costs are principally based on manufacturer's recommendations, and component replacement and cost. HRC also used data from operating mines of similar size in developing the operating costs for the mine. The cost details by department and category over the life of the mine is shown in Table 21-13.

Table 21-13: Mine Operating Costs (USD)

Department	Average Yearly Costs	$/ton Mined	$/ton Moved
Mine G&A	$639,916	$0.17	$0.15
Drilling	$1,265,510	$0.34	$0.30
Blasting	$1,410,661	$0.37	$0.34
Loading	$1,225,039	$0.33	$0.29
Haulage	$2,610,524	$0.69	$0.62
Roads & Dumps	$1,789,726	$0.48	$0.43
Dewatering	$24,944	$0.01	$0.01
Mine Mtce.	$1,181,102	$0.31	$0.28
Engineering	$469,883	$0.13	$0.12
Geology	$599,703	$0.16	$0.14
Grand Total	**$11,217,007**	**$2.99**	**$2.69**

Table 21-14 shows the distribution of the estimated mine operating costs by cost category for a typical year. On average the mine operating costs are estimated $2.99 per ton of material mined or $2.69 per ton of material moved.

Table 21-14: Mine Operating Costs by Category (USD)

Category	Average Yearly Costs	$/ton Mined	$/ton Moved
Labor & Benefits	$5,406,364	$1.44	$1.29
Fuel & Lubes	$2,633,521	$0.70	$0.63
Tires	$566,678	$0.15	$0.14
Repair Parts	$479,134	$0.13	$0.11
Wear parts	$515,302	$0.14	$0.12
Materials/Supplies	$114,918	$0.03	$0.03
Blasting/Supplies	$973,808	$0.26	$0.23
Services	$491,607	$0.13	$0.12
Energy	$4,314	$0.00	$0.00
Lease	$6,163	$0.00	$0.00
Training/travel	$25,197	$0.01	$0.01
Grand Total	**$11,217,007**	**$2.99**	**$2.69**

Mine operating labor rates and burdens percentages are based on industry averages and data from Idaho Department of Labor on average labor rates in the region. Mine staffing levels and rates for the life of the reserve are shown in Table 21-15. Mine manpower levels are mainly driven by equipment requirements with a maximum number of people required in year 7 at 89 and the average life of the mine requirement is estimated at 73 people.

Table 21-15: Manpower Requirements for Mine Operations

Mine Manpower			Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Mining G&A	Rate	Burden									
Mine Manager	$120,000	31%	1	1	1	1	1	1	1	1	1
Mine Foreman	$75,000	32%	4	4	4	4	4	4	4	4	4
Maintenance Superintendent	$110,000	31%	1	1	1	1	1	1	1	1	1
Total Mining G&A			6	6	6	6	6	6	6	6	6
Drilling and Blasting											
Driller	$26.50	34%	4	4	4	4	4	4	4	4	3
Blaster	$28.00	34%	2	2	2	2	2	2	2	2	2
Blaster Helper	$23.00	34%	1	1	1	1	1	1	1	1	1
Total Drilling and Blasting			7	7	7	7	7	7	7	7	6
Loading											
Excavator/Loader Operator	$29.00	33%	4	4	4	5	8	8	8	8	7
Total Loading			4	4	4	5	8	8	8	8	7
Hauling											
Truck Driver	$24.00	34%	12	10	11	14	20	16	18	24	20
Total Haulage			12	10	11	14	20	16	18	24	20
Roads & Dumps											
Dozer Operator	$26.50	34%	4	4	4	4	4	4	4	4	4
Grader Operator	$24.70	34%	4	4	4	4	4	4	4	4	3
Total Roads & Dumps			8	8	8	8	8	8	8	8	7
Mine Mtce.											
Lead Mechanic	$31.00	33%	4	4	4	4	4	4	4	4	4
Heavy Equipment Mechanic	$30.00	33%	5	4	4	5	8	6	7	10	8
Mechanic	$28.50	34%	2	2	2	2	2	2	2	2	2
Welder/Mechanic	$26.00	34%	2	2	2	2	2	2	2	2	2
Apprentice/Fueler	$26.00	34%	5	4	4	5	8	6	7	10	8
Planner	$24.75	34%	1	1	1	1	1	1	1	1	1
Total Mine Mtce.			19	17	18	20	25	22	23	28	25
Total Mine OP Operations			56	53	53	60	74	67	70	81	70
Engineering											
Sr Mining Engineer	$110,000	31%	1	1	1	1	1	1	1	1	1
Junior Engineer	$75,000	32%	1	1	1	1	1	1	1	1	1
Surveyor	$26.00	34%	2	2	2	2	2	2	2	2	2
Total Engineering			4	4	4	4	4	4	4	4	4
Geology											
Sr Geologist	$105,000	31%	1	1	1	1	1	1	1	1	1
Geologist	$75,000	32%	1	1	1	1	1	1	1	1	1
Sampler	$25.00	34%	2	2	2	2	2	2	2	2	2
Total Geology			4	4	4	4	4	4	4	4	3
Total Mine Eng & Geo			8	8	8	8	8	8	8	8	7
Total Mine Department			64	61	61	68	82	75	78	89	77

21.6.2 Process and Infrastructure Operating Costs

Final plant operating cost estimates were $19.3 million per year, giving a cost of $13.78 per short ton as shown in Table 21-16.

Table 21-16: Estimated Processing Plant Operating Costs (USD)

Operating Cost	Total Cost ($)	$/Short Ton	% of Total Cost
Labor	2,965,040	2.12	15.4%
Power	2,592,863	1.85	13.4%
Fuel	516,012	0.37	2.7%
General Maintenance, @3%	825,265	0.59	4.3%
Reagents	8,407,350	6.01	43.6%
Consumables	3,983,000	2.84	20.6%
Total	**19,289,530**	**13.78**	**-**

21.6.2.1 Cost Distribution

The distribution of operating costs was not unexpected for this scale operations, with the four main operating cost expenditures in descending order being:

- Reagents and Consumables;
- Labor;
- Power; and
- Maintenance

G&A costs have been covered under the mining operating costs. Items of expenditure higher than normally expected for copper mining operations related specifically to sulphuric acid consumption.

21.6.2.2 Labor

Estimated labor costs were developed by a build-up of base labor rates, burden and required work force numbers.

Workforce numbers were developed using a bottom-up approach by assessing requirements in each area, and from the internal body of knowledge.

The labor cost estimate was prepared by developing lists of hourly and salaried employees that will be required for mill operations. The mill is expected to run on a two (12 hour) shift per day basis. The day shift normally will have 18 people and night shift 13. An additional 13 hourly employees were added to account for replacements for personnel on sick or personal leave or otherwise absent bringing the total number of hourly employees to 43. Hourly wage rates were assigned based on job classification and local prevailing wages; these rates do not include shift differentials. A burden (employer paid benefits, taxes, insurance, etc.) of 38% was added to those wages. The annual hourly employee cost is estimated at $2,674,776.

The estimate includes 8 salaried employees. Again, salaries are estimated based on position and prevailing salaries predicted for the location. The 38% burden is also added to the salaries. The predicted annual salaried employee cost is $847,400.

The total annual personnel cost for the concentrator is $2,965,040 which translates to $2.12 per short ton.

21.6.2.3 Power Requirements

An entire load list was developed for the project. Power consumption was estimated for each piece of equipment between budgetary quotes, internal body of knowledge, and similar projects.

Power consumption for significant items were detailed in Process Plant Operating Costs. The major equipment detailed include but are not limited to:

- Crushers;
- Ball Mill;
- Flotation Equipment;
- Thickeners;
- Filters;
- Water Treatment Equipment;
- Ball mill;
- Pumps; and
- Conveyors

For smaller equipment a line item was entered to capture the remaining power consumption for the processing plant and infrastructure.

The installed motor power for each piece of equipment and ancillary items (lighting, HVAC, etc.) was totaled at 6,652 kW. The power consumption was estimated at 80% of the installed capacity. The total estimated power consumption is approximately 48 GWh/year. Local power cost is $0.058/kWh, for a total annual power cost of $2,592,863 or $1.85 per short ton.

21.6.2.4 Fuel

Fuel consumption estimates were developed for each item of process plant mobile equipment, by estimating annual operating hours and using vendor documented or estimated fuel consumptions for each equipment item. The cost of fuel, encompassing both diesel and gasoline, is currently estimated to be $3 per gallon.

Fuel cost for mobile equipment were estimated at $516,012 per year or $0.37 per short ton.

21.6.2.5 Maintenance

Annual maintenance costs were estimated at 3% of the new cost of the equipment in each area.

Large wear items (crusher wear liners, ball mill lifters / liners, belts, screen decks) were identified and listed separately in the consumables section. Maintenance cost estimates were $825,265 per year or $0.59 per short ton.

21.6.2.6 Reagents

Reagent costs were estimated by applying the determined consumption rates with the cost of delivered reagents to site.

Consumption rates for major reagents from test work and assumptions are listed below:

- Consumption of flotation reagents for sulfide rougher flotation: 0.5 lb/ton.
- Consumption of flotation reagents for carbonate flotation: 0.5 lb/ton.
- Consumption of flotation reagents for sulfide cleaner flotation: 0.003 lb/ton.
- Consumption of sulphuric acid for leaching: 50 lb/ton.

Reagent costs were developed from internal body of knowledge and similar projects. Reagent costs were estimated to be $8,407,350 per year or $6.01 per short ton.

21.6.2.7 Consumables

Consumable costs were estimated by calculating or estimating consumable consumption rates coupled with quotes or estimates for unit prices.

Consumables were estimated for the following equipment:

- Grinding Media and Liners for the Ball Mill: 1.0 lb/ton
- Shredded Iron for Cementation: 7.0 lb/ton
- Crusher Liners, Screen Decks, and Conveyor Belts: estimated as a dollar value per year.

Consumable costs were based upon budgetary quotes, internal body of knowledge, and similar projects. Consumable costs were estimated to be $3,983,000 per year or $2.84 per short ton.

21.6.2.8 Water Treatment Plant

The carbonate flotation concentrate, comprising primarily of lime will be pumped to the water treatment plant (WTP). This lime content allows the effluent of the plant to achieve a neutral pH. Treatment of the excess effluent to neutralize the acid will allow that water to be re-used in the flotation section of the concentrator. The projected treatment involves a straightforward lime neutralization process aimed at elevating the pH level above 7, which is expected to be adequate for the reusability of process water. Consequently, the only associated costs with the WTP are those for power and general maintenance. Power consumption for the WTP was included in Section 21.6.2.3. General maintenance for the WTP was included in 21.6.2.5.

21.6.2.9 Tailings Storage Facilities

The tailings operating costs encompass various elements, notably the conveyance of tailings to the Tailings Storage Facility (TSF), shaping and compaction of the TSF, and pumping from the tailings pond to the processing facility. These expenditures have been accounted for in preceding sections, with mobile equipment fuel costs delineated in Section 21.6.2.4, power consumption expenses outlined in Section21.6.2.3, and labor costs detailed in Section 21.6.2.2.

21.6.3 Process Plant Inputs

21.6.3.1 Process Plant Labor

The labor breakdown for the process plant shown in Table 21-17 represents the personnel required at steady state operations.

Table 21-17: Estimated Process Plant Personnel and Labor Rates (USD)

Position	Day Shift	Night Shift	Off	Total Personnel	Base Salary	Base Wage ($/hour)	Total Wage ($/year)	Burden @38% ($/year)	Total Cost ($/year)
Administration – Not included									
Process Plant – Hourly									
Control Room Operators	0	0	0	0	-	25	52,829	20,075	0
Crusher Operators	2	2	2	6	-	24	49,603	18,849	410,717
Grinding Operators + Flotation	2	2	2	6	-	23	46,800	17,784	387,504
Leach Operators + Disc Filter +Cementation	2	2	2	6	-	20	42,128	16,008	348,816
Assayers (Mackay)	1	0	0	1	-	24	50,854	19,325	70,179
Chemist (Mackay)	2	0	0	2	-	24	50,854	19,325	140,358
Prep Tech (Mackay)	1	0	0	1	-	18	36,928	14,032	50,960
Samplers (Plant)	2	0	0	2	-	18	36,928	14,032	101,920
Laborers (Plant)	2	1	1	4	-	17	35,782	13,597	197,517
Mechanics (Plant)	1	1	1	3	-	24	49,694	18,884	205,733
Mobile Equipment Operator	1	0	0	1	-	24	49,694	18,884	68,578
Electricians (Plant)	1	1	1	3	-	26	53,372	20,281	220,958
Subtotal	**17**	9	9	35	**-**	**-**	**1,596,551**	**606,689**	**2,203,240**
Process Plant – Salaried									
Mill Superintendent (Mackay)	1	-	-	1	90,000	-	90,000	34,200	124,200
General Foremen	0	-	-	0	90,000	-	90,000	34,200	0
Maintenance Foreman (Mackay)	1	-	-	1	92,971	-	92,971	35,329	128,300
Plant Foreman (Plant)	1	1	-	2	80,000	-	80,000	30,400	220,800
Senior Metallurgist	0	-	-	0	92,029	-	92,029	34,971	0
Metallurgist (Mackay)	1	-	-	1	85,000	-	85,000	32,300	117,300
Process Technician	1	-	-	1	62,029	-	62,029	23,571	85,600
Instrument Technician (Plant)	1	-	-	1	62,029	-	62,029	23,571	85,600
Process Foreman	0	-	-	0	70,000	-	70,000	26,600	0
Subtotal	6	1	-	**7**	**-**	**-**	**552,029**	**209,771**	**761,800**
Labor - Subtotal	**25**	**10**	**9**	**42**	**-**	**-**	**2,148,580**	**816,460**	**2,965,040**

21.6.3.2 Reagents

Reagent Consumption rates and costs are shown Table 21-18. Consumption rates are based upon test metallurgical test work and prices are based on internal body of knowledge and similar projects.

Table 21-18: Process Reagents

Reagent Use	Reagents	Consumption Rate	Reagent Cost ($) per lb
Sulphide Rougher Flotation	PAX, AP404, AP208, AF-65	0.50 lb/ton	1.75
Carbonate Flotation	Fatty Acid	0.50 lb/ton	1.25
Sulphide Cleaner Flotation	PAX, AP404, AP208, AF-65	0.003 lb/ton	1.75
Acid Leaching	Sulphuric Acid	60 lb/ton	0.08

21.6.3.3 Consumables

Consumable consumption rates and costs are shown in Table 21-19. Consumption rates and costs are based upon benchmark data and internal body of knowledge given the ores processed at the site.

Table 21-19: Process Consumables

Consumable	Consumption Rate	Consumable Cost Per Unit
Jaw Crusher	1 – liner per year	$10,000 per set
Secondary Cone Crusher	4 – Liners per year	$5,000 per set
Vibrating Screen	4 – Screen decks per year	$2,000 per set
Tertiary Crusher	4 – liners per year	$5,000 per set
Fine Ore Bind Feeder & Conveyor	1 – Belt/Liner	$5,000 per set
Conveyors	7 – Belts	$5,000 per set
Grinding Media & Ball Mill Liners	1 lb/ton	$0.90 per lb
Shredded Iron	7 lb/ton	$0.25 per lb

21.6.4 General and Administrative Operating Costs

The G&A costs have been developed from the QP's knowledge and experience as well as data from similar size operations. The major G&A cost component is staff and labor, but G&A also covers such things as security, office equipment, heat and lighting, communications, overtime, property insurance, office supplies, computer system license fees, admin building maintenance, janitorial services, outside services and allowances for travel and meetings. The costs details by department over the life of the mine are shown in Table 21-21. On average the G&A operating costs are estimated to be $1.87 per ton of mineralized material milled as presented in Table 21-20.

Table 21-20: G&A Operating Costs (USD)

Department	Average Yearly Costs	$/ton Ore
Admin	$1,388,541	$0.99
Human Relations	$221,971	$0.16
Accounting	$176,240	$0.13
Environmental	$310,892	$0.22
Purchasing	$174,525	$0.12
Security & Safety	$345,888	$0.25
Grand Total	**$2,618,057**	**$1.87**

G&A operating labor rates and burdens percentages are based on industry averages and data from Idaho Department of Labor on average labor rates in the region. G&A staffing levels and rates for the life of the mine are shown in Table 21-21. The average personnel estimated for G&A is 15 people, the total property requirements for the life of the mine are also shown in Table 21-21. Total property personnel are estimated to average 141 people over the life of the mine with a maximum of 156 people.

Table 21-21: G&A Operating Manpower

G&A Manpower			Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
G&A	Rate	Burden								
General Manager	$130,000	31%	1	1	1	1	1	1	1	1
Controller	$75,000	32%	1	1	1	1	1	1	1	1
Accounts Payable Clerk	$21.00	35%	1	1	1	1	1	1	1	1
IT Coordinator	$25.00	34%	1	1	1	1	1	1	1	1
Environmental Manager	$95,000	31%	1	1	1	1	1	1	1	1
Environmental Tech	$65,000	32%	1	1	1	1	1	1	1	1
Safety Manager	$85,000	31%	1	1	1	1	1	1	1	1
Human Resources Manager	$85,000	31%	1	1	1	1	1	1	1	1
HR Clerk	$24.00	34%	1	1	1	1	1	1	1	1
Security Guards	$18.00	36%	4	4	4	4	4	4	4	3
Purchasing Manager	$70,000	32%	1	1	1	1	1	1	1	1
Purchasing Agent	$21.50	35%	1	1	1	1	1	1	1	1
Total G&A			**15**	**15**	**15**	**15**	**15**	**15**	**15**	**14**
Total Mine Department			**61**	**61**	**68**	**82**	**75**	**78**	**89**	**77**
Total Process Department			**52**	**52**	**52**	**52**	**52**	**52**	**52**	**52**
Total Property			**128**	**128**	**135**	**149**	**142**	**145**	**156**	**144**

22. ECONOMIC ANALYSIS

The Project has been evaluated using a constant US dollar, after-tax discounted cashflow methodology. This valuation method requires projecting material balances estimated from operations and calculating the economic analysis. Cashflows are calculated from sales of metal plus net equipment salvage value, less cash outflows such as operating costs, capital costs, working capital changes, any applicable taxes and reclamation costs. Resulting annual cash flows are used to calculate the NPV and IRR of the Project. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations, and as such, the actual post-tax results may differ from those estimated. Information contained and certain statements made herein are considered forward-looking within the meaning of applicable Canadian securities laws. These statements address future events and conditions and so involve inherent risks and uncertainties. Actual results could differ from those currently projected.

22.1 Summary

This Pre-feasibility Study (PFS) for the Empire Project evaluates processing 11.1 million tons of ore at a milling rate of 4,000 tpd (1.4 million tons per annum) for eight years. The processing operations are planned with a mill and flotation circuit to produce a copper concentrate, with contained payables for silver, gold, and copper followed by a cementation circuit to produce precipitate copper which will only be payable for copper. The mine plan is based on an owner operated open pit operation. During the mining of pit ore, the peak mineralized material and waste production is estimated at 11,500 t/d and an average production rate of 10,000 t/d. The life of mine average copper grade is estimated to be 0.49% for copper, 0.009 opt for gold and 0.42 opt for silver. The average life of mine stripping ratio is estimated to be 1.69:1 for the open pit.

The economic analysis of the base case scenario for the Project uses metal prices of $4.45/lb for copper, $2,325/oz for gold and $27.25/oz for silver. The economic model shows an Pre-Tax Net Present Value @ 7.5% (“NPV-7.5”) of $87.86 million using, as well as an Pre-Tax Internal Rate of Return (“IRR”) of 46.4%. Table 22-1 summarizes the projected Cashflow; Net Present Value at varying rates; Internal Rate of Return; years of positive cash flows to repay the negative cash flow (“payback period”); multiple of positive cash flows compared to the maximum negative cash flow (“payback multiple”) for the Project on both After-Tax and Before-Tax bases.

Table 22-1: Summary of Economic Results (USD)

Project Evaluation Overview	After Tax	Before Tax
Cumulative Net Cashflow (millions)	$132.44	$152.98
NPV @ 5.0%; (millions)	$89.55	$105.44
NPV @ 7.5%; (millions)	$73.75	$87.86
NPV @ 10.0%; (millions)	$60.71	$73.29
Internal Rate of Return	40.2%	46.4%
Payback Period	1.66	1.41
Payback Multiple	2.92	3.21
Benefit Cost Ratio	7.61	8.87
Initial Capital (millions)	$62.60	$62.60
Max Neg. Cashflow (millions)	-$69.09	-$69.09

22.2 Prices and Revenue

The metal prices for the base case evaluation use metal prices of $4.45/lb for copper, $2,325/oz for gold and $27.25/oz for silver. Konnex has not yet entered into discussions with potential smelter concentrate trades so for the PFS standard industry terms have been used for the evaluation. The metal prices and smelter terms are presented below in Table 22-2.

Table 22-2: Metal Prices and Smelter Terms

Reserve Parameter	Units	Cu	Au	Ag
Commodity Prices	$/oz or $/lb	$4.45	$2325	27.25
Flotation Process Recoveries				
Flotation _ Cu Concentrate	%	24.50%	40.00%	40.00%
Concentrate (Payables)				
Flotation_ Cu Concentrate (Au Payable based on grade)	%	95.00%	90-97%	95.00%
Cementation Process Recoveries				
Cementation (Total Copper Recovery after Flotation)	%	83.00%	0.00%	0.00%
Cementation Payable	%	100.00%	0.00%	0.00%
Treatment/Refining Charges				
Copper Con. Refining	Ag $/oz	$0.40		
Copper Con. Refining	Au $/oz	$4.00		
Copper Con. Trucking & Shipping $/t conc	wet	$80.00		
Copper Con. Treatment $/t conc	wet	$90.00		
Copper Cementation Shipping $/lb	Cu $/lb	$0.04		
Copper Cementation Selling $/lb	Cu $/lb	$0.02		

Table 22-3 shows the estimated LOM metal production and payable metal from the copper concentrate and the copper from the cementation circuit. 46% of the revenue is estimated to come from the copper concentrate with 54% coming from the copper cementation. Overall, 75% of the revenue is estimated from copper, 16% from gold and 9% from silver.

Table 22-3: Metal Production and Net Revenue Estimate

LOM - Metal Production and Net Revenue	
Copper Produced Float (lbs.)	26,688,923
Copper Produced Cementation (lbs.)	63,740,228
Total Copper Produced (lbs.)	90,429,151
Gold Produced Float (oz)	41,403
Silver Produced Float (oz)	1,852,334
Copper Payable Float (lbs.)	25,354,477
Copper Payable Cementation (lbs.)	63,740,228
Total Copper Payable (lbs.)	89,094,705
Gold Payable Float (oz)	40,161
Silver Payable Float (oz)	1,759,717
	(USD $000s)
Gross Revenue Float	$254,155
Gross Revenue Cementation	$283,644
Total Gross Revenue	$537,799
Smelter and Transportation Charges	-$13,660
Net Revenue	$524,138
Royalties	-$20,179
Net Revenue	**$503,959**

22.3 Capital Costs

LoM capital requirements are estimated at $83.9 million as summarized in Table 22-4. Initial capital for this study includes working capital and items purchased in the first two years before production and is estimated to be $62.6 million. Total capital carried in the financial model is all for new equipment and construction excluding the ball mills which have already been purchased by Konnex.

Table 22-4: Estimated Capital Cost Summary (USD $000s)

Capital Costs ($000s)	Initial	Sustaining	Total LOM
2000 Mine Infrastructure and Equipment	$6,611	$16,409	$23,020
3100 Crushing and Conveying	$4,327	$0	$4,327
3200-3400, 3520 & 3530 Process Plant	$22,766	$0	$22,766
4000 Infrastructure and Utlilites	$9,728	$0	$9,728
3540 Tailings and Ponds	$9,742	$6,474	$16,216
Total Direct Costs	$53,173	$22,883	$76,056
8000 Owner Costs, EPCM, Commissioning	$6,760	$0	$6,760
9100 - 9400 First Fills, Spares	$2,666	$0	$2,666
8500 Reclamation & Bonding	$0	$1,300	$1,300
9700 Contingency	$0	$0	$0
10000 Salvage Value	$0	($2,903)	($2,903)
Total Indirect Costs	$9,427	($1,603)	$7,824
TOTAL Capital	$62,600	$21,280	$83,880
9500 Working Capital	$891	($891)	$0
TOTAL CAPITAL/Working Capital	**$63,491**	**$20,389**	**$83,880**

22.4 Operating Costs

The operating costs include the ongoing cost of operations related to mining, processing, tailings disposal and general administration activities. The operating costs were determined based on Fortes's and HRC's in-house database of projects and studies and budgetary quotes or estimates for consumables and supplies. Following the AISC guidelines, LOM average Cash Operating Cost is projected to be $2.27/lb of CuEq sold. The LOM average base case Total Cash Costs (including royalties) is expected to be $2.44/lb CuEq. The total AISC summary per ton of mill feed and per pound of CuEq sold is expected to be $33.35/t and $3.15/lb respectively as shown in Table 22-5.

Table 22-5: Total Operating Cost Summary (USD)

Operating Costs	$/CuEq lb	$/ton ore	$/ton mined	$/ton moved
Total Mining	$0.76	$8.05	$2.99	$2.69
Total Processing	$1.30	$13.78		
Total Site General & Administration	$0.18	$1.87		
Mine License & Property Tax	$0.03	$0.30		
Cash Operating Costs	**$2.27**	**$24.00**		
Royalties	$0.17	$1.81		
Total Cash Costs	**$2.44**	**$25.81**		
Capital	$0.71	$7.54		
All-in Sustaining Costs	**$3.15**	**$33.35**		

The average yearly mine operating costs are estimated to be $11.2M which is equivalent to $2.99/ton mined and $2.69/ton moved. The costs details by department over the life of the mine are shown in Table 22-6 and Figure 22-1 shows the distribution of these costs by department.

Table 22-6: Mine Operating Costs (USD)

Department	Average Yearly Costs	$/ton Mined	$/ton Moved
Mine G&A	$639,916	$0.17	$0.15
Drilling	$1,265,510	$0.34	$0.30
Blasting	$1,410,661	$0.37	$0.34
Loading	$1,225,039	$0.33	$0.29
Haulage	$2,610,524	$0.69	$0.62
Roads & Dumps	$1,789,726	$0.48	$0.43
Dewatering	$24,944	$0.01	$0.01
Mine Mtce.	$1,181,102	$0.31	$0.28
Engineering	$469,883	$0.13	$0.12
Geology	$599,703	$0.16	$0.14
Grand Total	**$11,217,007**	**$2.99**	**$2.69**

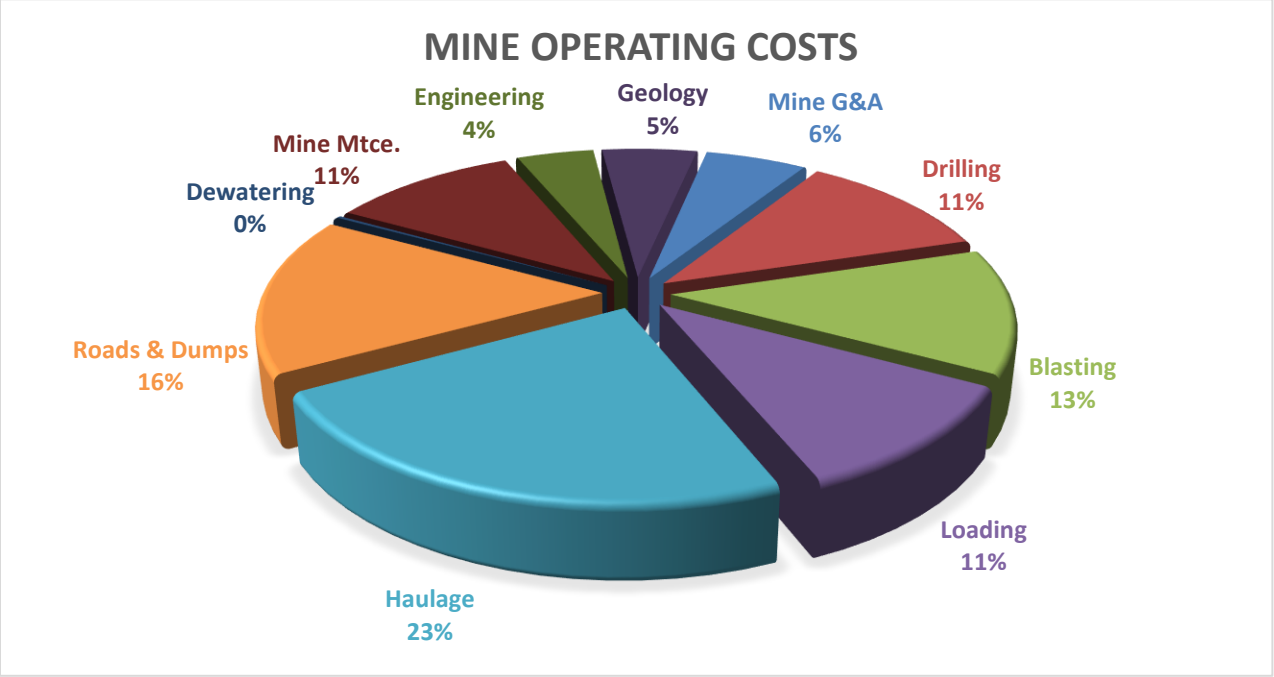


Figure 22-1: Mine Operating Cost Distribution

The average yearly processing operating costs are estimated to be $19.3M which is equivalent to $13.78/ton milled. The costs details by department over the life of the mine are shown in Table 21-7 and Figure 22-2 shows the distribution of these costs by department.

Table 22-7: Process Operating Costs (USD)

Department	Average Yearly Costs	$/ton Ore
Mill G&A	$997,790	$0.71
Crushing	$812,729	$0.58
Grinding	$4,174,280	$2.98
Flotation	$2,640,874	$1.89
Leaching	$9,325,316	$6.66
Tailings	$243,748	$0.17
Mill Maintenance	$731,376	$0.52
Laboratory	$363,418	$0.26
Grand Total	**$19,289,530**	**$13.78**

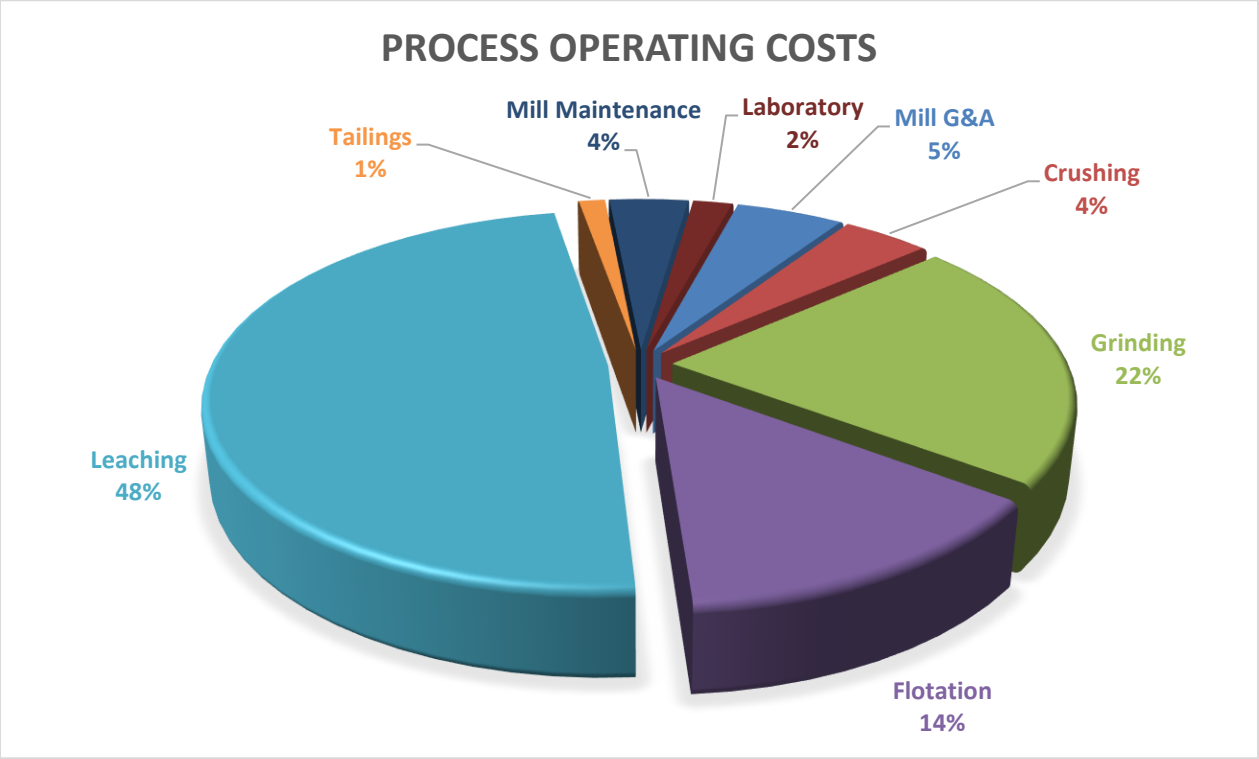


Figure 22-2: Process Operating Cost Distribution

The average yearly G&A operating costs are estimated to be $2.6M which is equivalent to $1.87/ton milled. The costs details by department over the life of the mine are shown in Table 22-8 and Figure 22-3 shows the distribution of these costs by department.

Table 22-8: G&A Operating Costs (USD)

Department	Average Yearly Costs	$/ton Ore
Admin	$1,388,541	$0.99
Human Relations	$221,971	$0.16
Accounting	$176,240	$0.13
Environmental	$310,892	$0.22
Purchasing	$174,525	$0.12
Security & Safety	$345,888	$0.25
Grand Total	**$2,618,057**	**$1.87**

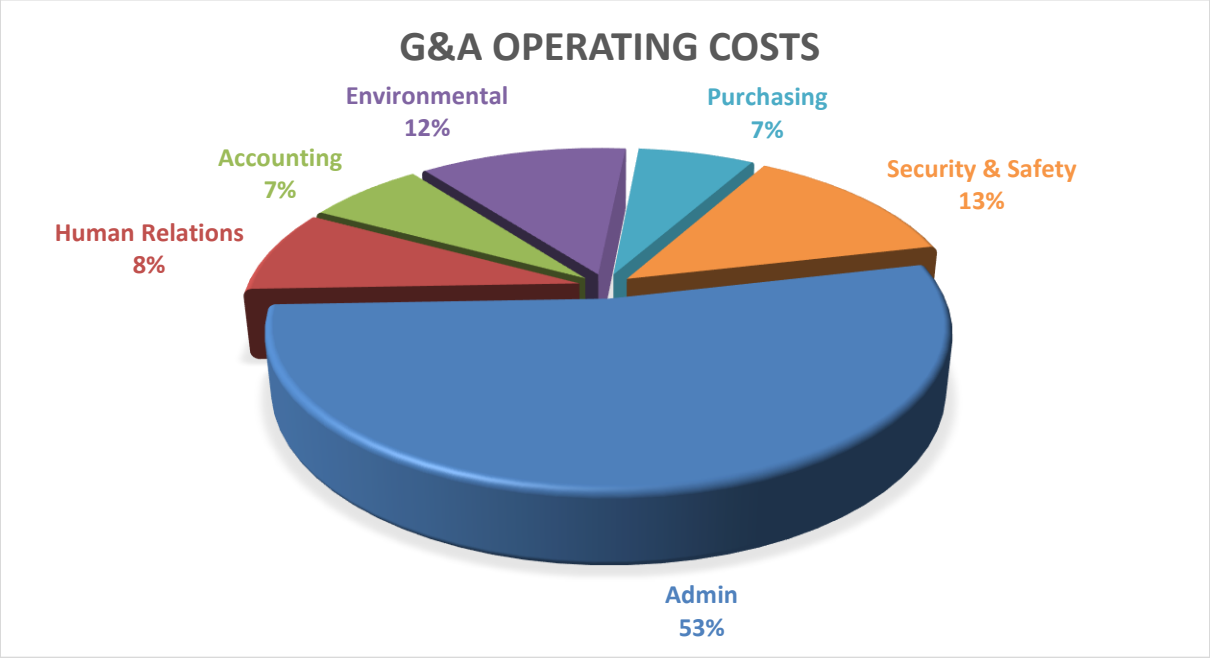


Figure 22-3: G&A Operating Cost Distribution

22.5 Taxes

State, local, and federal taxes, including income taxes and the Idaho Mining License tax, have been considered in this study, and are included in the economic analysis.

22.6 Royalties

A 3.85 percent royalty, calculated on the gross proceeds less transportation and refining costs, has been included for all of the metal produced, as required by underlying agreements. The royalty includes a 2.5% royalty held by ExGen Resources Inc. and 90% of the reserves are within the Mackay LLC claim which hold a 1.5% royalty.

22.7 Financial Model

The economic analysis was done on monthly basis but summarized into an annual model; the annual production and cashflow model is shown below in Table 22-9.

Table 22-9: Production Schedule and Cashflow Model

	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	
3836 tpd 0, $4.45 Copper Price	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Life-of-Mine
Note: All Tons and $ Values are in 1,000's	Total	TOTAL	TOTAL	TOTAL	TOTAL	TOTAL	TOTAL	TOTAL	TOTAL	TOTAL	TOTAL	Yr -2 - Yr 9
PRODUCTION SCHEDULE												
TOTAL MINE PRODUCTION												
Pit High Grade Ore	0	80	1,254	1,453	1,367	1,173	1,348	1,409	970	687	0	9,740
Grade: CuT (%)	-	0.54	0.67	0.59	0.62	0.44	0.42	0.40	0.48	0.69	-	0.53
Grade: Zn (%)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Grade: Au (oz/t)	0.000	0.005	0.012	0.010	0.007	0.013	0.008	0.009	0.007	0.014	0	0.010
Grade: Ag (oz/t)	0.00	0.73	0.72	0.49	0.45	0.44	0.29	0.25	0.37	0.62	0	0.44
Pit Low Grade Ore	0	7	53	89	84	238	313	328	232	46	0	1,390
Grade: CuT (%)	-	0.27	0.19	0.24	0.23	0.20	0.21	0.20	0.21	0.12	-	0.20
Grade: Zn (%)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Grade: Au (oz/t)	0.000	0.001	0.006	0.004	0.004	0.008	0.006	0.008	0.007	0.012	0	0.007
Grade: Ag (oz/t)	0.00	0.38	0.46	0.31	0.28	0.18	0.19	0.18	0.19	0.37	0	0.22
Pit Waste - Rock	0	615	1,250	1,238	1,988	2,743	2,391	2,475	3,891	2,215	0	18,806
Alluvium	0	0	0	0	0	0	0	0	0	0	0	0
Ug Waste	0	0	0	0	0	0	0	0	0	0	0	0
Total Tons Mined	**0**	**702**	**2,556**	**2,780**	**3,439**	**4,153**	**4,051**	**4,212**	**5,093**	**2,948**	**0**	**29,936**
Other, Stemming, Roadbed, Rehandle etc..	0	80	268	243	240	448	293	240	664	874	0	3,350
Total Tons Moved	**0**	**782**	**2,825**	**3,024**	**3,679**	**4,601**	**4,344**	**4,452**	**5,757**	**3,822**	**0**	**33,287**
Strip Ratio	0.00	7.04	0.96	0.80	1.37	1.94	1.44	1.42	3.24	3.02	0	1.69
Tons ore mined/day	0.0	0.2	3.4	4.0	3.7	3.2	3.7	3.9	2.7	1.9	0	
Tons Heap mined/day	0.0	0.0	0.1	0.2	0.2	0.7	0.9	0.9	0.6	0.1	0.0	
Total Tons mined per day	0.0	1.9	7.0	7.6	9.4	11.4	11.1	11.5	14.0	8.1	0	
Copper Lbs Mined												
Pit HG Ore	0	870,373	16,722,100	17,078,307	16,841,014	10,381,128	11,224,206	11,296,745	9,346,247	9,481,695	0	103,241,814
Pit LG Ore	0	38,410	202,540	422,060	385,680	927,403	1,315,640	1,313,716	978,494	108,621	0	5,692,564
Total Contained Cu Lbs.	**0**	**908,782**	**16,924,640**	**17,500,367**	**17,226,694**	**11,308,531**	**12,539,846**	**12,610,461**	**10,324,741**	**9,590,315**	**0**	**108,934,379**
Tons to Mill												
Pit High Grade Ore	0	0	1,334	1,397	1,404	1,192	1,347	1,400	980	687	0	9,740
Grade: CuT (%)	-	-	0.66	0.58	0.62	0.45	0.42	0.40	0.48	0.69	-	0.53
Grade: Zn (%)	-	-	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-	0.000
Grade: Au (oz/t)	-	-	0.011	0.010	0.008	0.013	0.008	0.009	0.007	0.014	-	0.010
Grade: Ag (oz/t)	-	-	0.72	0.48	0.46	0.44	0.29	0.25	0.37	0.62	-	0.44

Pit Low Grade Ore	0	0	28	3	0	208	53	0	424	674	0	1,390
Grade: CuT (%)	-	-	0.23	0.21	-	0.21	0.21	-	0.21	0.20	-	0.20
Grade: Zn (%)	-	-	0.000	0.000	-	0.000	0.000	-	0.000	0.000	-	0.000
Grade: Au (oz/t)	-	-	0.004	0.006	-	0.007	0.006	-	0.007	0.007	-	0.007
Grade: Ag (oz/t)	-	-	0.41	0.25	-	0.22	0.20	-	0.20	0.22	-	0.22
Total Processed Tons	**0**	**0**	**1,362**	**1,400**	**1,404**	**1,400**	**1,400**	**1,400**	**1,404**	**1,360**	**0**	**11,130**
Total Cu%	-	-	**0.65**	**0.58**	**0.62**	**0.41**	**0.41**	**0.40**	**0.40**	**0.45**	-	**0.49**
Total Zn%	-	-	**0.000**	**0.000**	**0.000**	**0.000**	**0.000**	**0.000**	**0.000**	**0.000**	-	**0.000**
Total Au oz/t	-	-	**0.011**	**0.009**	**0.008**	**0.012**	**0.008**	**0.009**	**0.007**	**0.010**	-	**0.009**
Total Ag oz/t	-	-	**0.71**	**0.48**	**0.46**	**0.41**	**0.28**	**0.25**	**0.32**	**0.42**	-	**0.42**
Income Statement												
REVENUE												
Copper Produced Float (lbs.)	-	-	4,341,606	3,992,200	4,256,649	2,818,847	2,802,883	2,748,173	2,744,208	2,984,357	-	26,688,923
Copper Produced Cementation (lbs.)	-	-	10,368,906	9,534,433	10,166,007	6,732,154	6,694,027	6,563,366	6,553,897	7,127,437	-	63,740,228
Total Copper Produced (lbs.)	-	-	14,710,512	13,526,634	14,422,656	9,551,001	9,496,910	9,311,539	9,298,105	10,111,794	-	90,429,151
Gold Produced Float (oz)	-	-	5,982	5,317	4,275	6,558	4,365	5,165	4,064	5,679	-	41,403
Silver Produced Float (oz)	-	-	386,526	269,954	259,125	229,194	158,628	139,281	179,946	229,680	-	1,852,334
Copper Payable Float (lbs.)	-	-	4,124,525	3,792,590	4,043,817	2,677,904	2,662,738	2,610,764	2,606,998	2,835,139	-	25,354,477
Copper Payable Cementation (lbs.)	-	-	10,368,906	9,534,433	10,166,007	6,732,154	6,694,027	6,563,366	6,553,897	7,127,437	-	63,740,228
Total Copper Payable (lbs.)	-	-	14,493,432	13,327,024	14,209,824	9,410,058	9,356,766	9,174,131	9,160,895	9,962,576	-	89,094,705
Gold Payable Float (oz)	-	-	5,802	5,157	4,147	6,361	4,234	5,010	3,942	5,509	-	40,161
Silver Payable Float (oz)	-	-	367,200	256,456	246,169	217,734	150,697	132,317	170,948	218,196	-	1,759,717
Gross Revenue Float	$0	$0	$41,850	$35,856	$34,345	$32,639	$25,799	$26,871	$25,426	$31,370	$0	$254,155
Gross Revenue Cementation	$0	$0	$46,142	$42,428	$45,239	$29,958	$29,788	$29,207	$29,165	$31,717	$0	$283,644
Total Gross Revenue	$0	$0	$87,992	$78,284	$79,583	$62,597	$55,587	$56,078	$54,590	$63,087	$0	$537,799
Smelter and Transportation Charges	$0	$0	$2,252	$2,037	$2,156	$1,464	$1,421	$1,391	$1,400	$1,540	$0	$13,660
Net Revenue	$0	$0	$85,740	$76,247	$77,427	$61,133	$54,166	$54,687	$53,191	$61,546	$0	$524,138
Royalties	$0	$0	$3,301	$2,936	$2,981	$2,354	$2,085	$2,105	$2,048	$2,370	$0	$20,179
Royalties	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Net Revenue	**$0**	**$0**	**$82,439**	**$73,312**	**$74,447**	**$58,779**	**$52,081**	**$52,582**	**$51,143**	**$59,177**	**$0**	**$503,959**

OPERATING EXPENSES												
Mining												
Heap Mining	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Mine G&A	$0	$203	$610	$611	$615	$617	$616	$616	$623	$577	$0	$5,087
Drilling	$0	$394	$1,233	$1,238	$1,232	$1,233	$1,229	$1,253	$1,236	$1,012	$0	$10,061
Blasting	$0	$329	$1,112	$1,170	$1,323	$1,488	$1,469	$1,508	$1,705	$1,111	$0	$11,215
Loading	$0	$234	$771	$805	$996	$1,404	$1,373	$1,395	$1,589	$1,184	$0	$9,750
Haulage	$0	$620	$1,596	$1,639	$2,102	$3,068	$2,532	$2,724	$3,652	$3,001	$0	$20,934
Roads & Dumps	$0	$571	$1,760	$1,760	$1,764	$1,760	$1,760	$1,760	$1,764	$1,535	$0	$14,433
Dewatering	$0	$8	$24	$24	$24	$24	$24	$24	$24	$20	$0	$198
Mine Mtce.	$0	$391	$1,144	$1,151	$1,157	$1,082	$1,089	$1,081	$1,059	$1,237	$0	$9,390
Engineering	$0	$151	$452	$453	$455	$455	$455	$455	$458	$402	$0	$3,736
Geology	$0	$172	$536	$546	$575	$604	$600	$607	$645	$481	$0	$4,768
Total Mining	**$0**	**$3,073**	**$9,239**	**$9,396**	**$10,244**	**$11,735**	**$11,147**	**$11,424**	**$12,755**	**$10,560**	**$0**	**$89,573**
Processing												
Float & Cementation	$0	$0	$18,770	$19,292	$19,345	$19,292	$19,292	$19,292	$19,345	$18,746	$0	$153,373
Total Processing	**$0**	**$0**	**$18,770**	**$19,292**	**$19,345**	**$19,292**	**$19,292**	**$19,292**	**$19,345**	**$18,746**	**$0**	**$153,373**
Site General & Administration												
Admin	$0	$391	$1,342	$1,342	$1,343	$1,342	$1,342	$1,342	$1,343	$1,252	$0	$11,039
Accounting	$0	$5	$175	$175	$175	$175	$175	$175	$175	$173	$0	$1,401
Purchasing	$0	$16	$171	$171	$172	$171	$171	$171	$172	$170	$0	$1,387
Environmental	$0	$32	$306	$306	$306	$306	$306	$306	$306	$298	$0	$2,472
Human Relations	$0	$35	$216	$216	$217	$216	$216	$216	$217	$213	$0	$1,765
Security & Safety	$0	$43	$343	$343	$344	$343	$343	$343	$344	$306	$0	$2,750
Total G&A	**$0**	**$522**	**$2,553**	**$2,553**	**$2,557**	**$2,553**	**$2,553**	**$2,553**	**$2,557**	**$2,412**	**$0**	**$20,814**
Idaho Mining License & Prop Tax	0	0	752	602	602	339	244	246	200	359	0	3,344
Operating Cost Contingency @ 0%	0	0	0	0	0	0	0	0	0	0	0	0
Cash Operating Costs	**$0**	**$3,595**	**$31,314**	**$31,843**	**$32,748**	**$33,919**	**$33,237**	**$33,515**	**$34,857**	**$32,076**	**$0**	**$267,104**
EBITDA	**$0**	**($3,595)**	**$51,126**	**$41,468**	**$41,699**	**$24,860**	**$18,844**	**$19,067**	**$16,286**	**$27,101**	**$0**	**$236,855**
Corporate Costs	0	0	0	0	0	0	0	0	0	0	0	0
Development Costs	0	0	0	0	0	0	0	0	0	0	0	0
Depreciation	0	0	8,899	13,017	11,432	10,695	9,905	9,677	9,623	12,235	0	85,483
Salvage Value	0	0	0	0	0	0	0	0	0	0	(2,903)	(2,903)
Reclamation Deduction	0	0	0	0	0	0	0	0	0	3,300	0	3,300
Equip. Interest Expense	0	0	0	0	0	0	0	0	0	0	0	0

Income - before NOL and Percentage Depletio	0	(3,595)	42,227	28,451	30,267	14,165	8,939	9,391	6,662	11,566	2,903	$150,975
Net Operating Loss Adjustment	0	3,595	(3,595)	0	0	0	0	0	0	0	0	0
Depletion	0	0	(18,647)	(13,733)	(14,609)	(6,837)	(4,315)	(4,533)	(3,216)	(5,583)	(1,401)	(72,874)
State Income Tax	0	0	(1,338)	(985)	(1,048)	(490)	(310)	(325)	(231)	(400)	(101)	(5,228)
Federal Income Tax	0	0	(3,916)	(2,884)	(3,068)	(1,436)	(906)	(952)	(675)	(1,172)	(294)	(15,304)
DPD Deduction	0	0	0	0	0	0	0	0	0	0	0	0
Taxable Income, less Tax	**$0**	**$0**	**$14,731**	**$10,849**	**$11,541**	**$5,401**	**$3,409**	**$3,581**	**$2,540**	**$4,410**	**$1,107**	**$57,570**
Cash Flow Calculation												
Adjustments for Non Cash Items												
Depreciation	$0	$0	$8,899	$13,017	$11,432	$10,695	$9,905	$9,677	$9,623	$12,235	$0	$85,483
Salvage Value	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	($2,903)	($2,903)
Reclamation Deduction	$0	$0	$0	$0	$0	$0	$0	$0	$0	$3,300	$0	$3,300
Net Operating Loss Adjustment	0	(3,595)	3,595	0	0	0	0	0	0	0	0	0
Depletion	0	0	18,647	13,733	14,609	6,837	4,315	4,533	3,216	5,583	1,401	72,874
Total Adjustments for Non Cash Items	**$0**	**($3,595)**	**$31,141**	**$26,750**	**$26,042**	**$17,532**	**$14,220**	**$14,209**	**$12,839**	**$21,118**	**($1,502)**	**$158,754**
Capital												
Acquisition of Reserves	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Initial Crushing & Conveying	$817	$3,510	$0	$0	$0	$0	$0	$0	$0	$0	$0	$4,327
Initial First Fills, Spares	504	2,163	0	0	0	0	0	0	0	0	0	2,666
Initial Tailings & Ponds	1,840	7,902	0	0	0	0	0	0	0	0	0	9,742
Initial Mine	511	6,100	0	0	0	0	0	0	0	0	0	6,611
Initial Process Plant	4,300	18,466	0	0	0	0	0	0	0	0	0	22,766
Initial Site General	2,191	4,570	0	0	0	0	0	0	0	0	0	6,760
Initial Infrastructure & Utilities	1,837	7,890	0	0	0	0	0	0	0	0	0	9,728
Initial Reclamation Bonding	0	2,000	0	0	0	0	0	0	0	0	(2,000)	0
Initial Contingency	0	0	0	0	0	0	0	0	0	0	0	0
Total Initial Capital	**$12,000**	**$52,600**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**($2,000)**	**$62,600**
Crushing & Conveying	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining Capital Contingency	0	0	0	0	0	0	0	0	0	0	0	0
Tailings & Ponds	0	0	0	0	6,474	0	0	0	0	0	0	6,474
Mine	0	0	1,243	1,270	2,595	2,504	1,991	2,041	2,793	1,972	0	16,409
Process Plant	0	0	0	0	0	0	0	0	0	0	0	0
Site General	0	0	0	0	0	0	0	0	0	0	0	0
Utilities	0	0	0	0	0	0	0	0	0	0	0	0
Reclamation	0	0	0	0	0	0	0	0	0	1,300	0	1,300
Contingency	0	0	0	0	0	0	0	0	0	0	0	0
Salvage Value	0	0	0	0	0	0	0	0	0	0	(2,903)	(2,903)

Total Capital	**$12,000**	**$52,600**	**$1,243**	**$1,270**	**$9,069**	**$2,504**	**$1,991**	**$2,041**	**$2,793**	**$3,272**	**($4,903)**	**$83,880**
Working capital	0	891	0	0	0	0	0	0	0	(891)	0	0
Equity Financing	0	0	0	0	0	0	0	0	0	0	0	0
Equipment Financing	0	0	0	0	0	0	0	0	0	0	0	0
Principal Payments	0	0	0	0	0	0	0	0	0	0	0	0
Total Capital/Working Capital	**$12,000**	**$53,491**	**$1,243**	**$1,270**	**$9,069**	**$2,504**	**$1,991**	**$2,041**	**$2,793**	**$2,381**	**($4,903)**	**$83,880**
CASHFLOW	**($12,000)**	**($57,086)**	**$44,629**	**$36,329**	**$28,514**	**$20,430**	**$15,637**	**$15,750**	**$12,586**	**$23,147**	**$4,508**	**$132,444**
CUMULATIVE CASHFLOW	**($12,000)**	**($69,087)**	**($24,458)**	**$11,872**	**$40,386**	**$60,816**	**$76,453**	**$92,203**	**$104,789**	**$127,936**	**$132,444**	**$132,444**

22.8 Sensitivities

22.8.1 Copper Price

The Project, like almost all precious metals projects, is very responsive to changes in the price of its chief commodity, copper. From the base case, a +10% change in the average copper price would change the NPV-7.5 by 29%, or approximately $21.1 million (Figure 22-4). Table 22-10 also shows the economic sensitivities, due to the change in copper price, in the Net Cash Flow, the Net Present Value at 7.5%, the Internal Rate of Return, the Payback Period, and the Payback Multiple.

Table 22-10: Copper Price Sensitivity Economic Results

Cu Price	Net Cash Flow, millions	NPV-7.5%, millions	IRR	Payback Period, Years	Payback Multiple
$3.12	$33.93	$9.67	12.7%	3.4	1.5
$3.56	$67.53	$31.47	23.0%	2.6	2.0
$4.01	$99.99	$52.62	31.9%	2.0	2.4
$4.45	**$132.44**	**$73.75**	**40.2%**	**1.7**	**2.9**
$4.90	$164.89	$94.89	47.9%	1.4	3.4
$5.34	$197.34	$116.02	55.2%	1.3	3.9
$5.79	$229.78	$137.15	62.3%	1.2	4.3

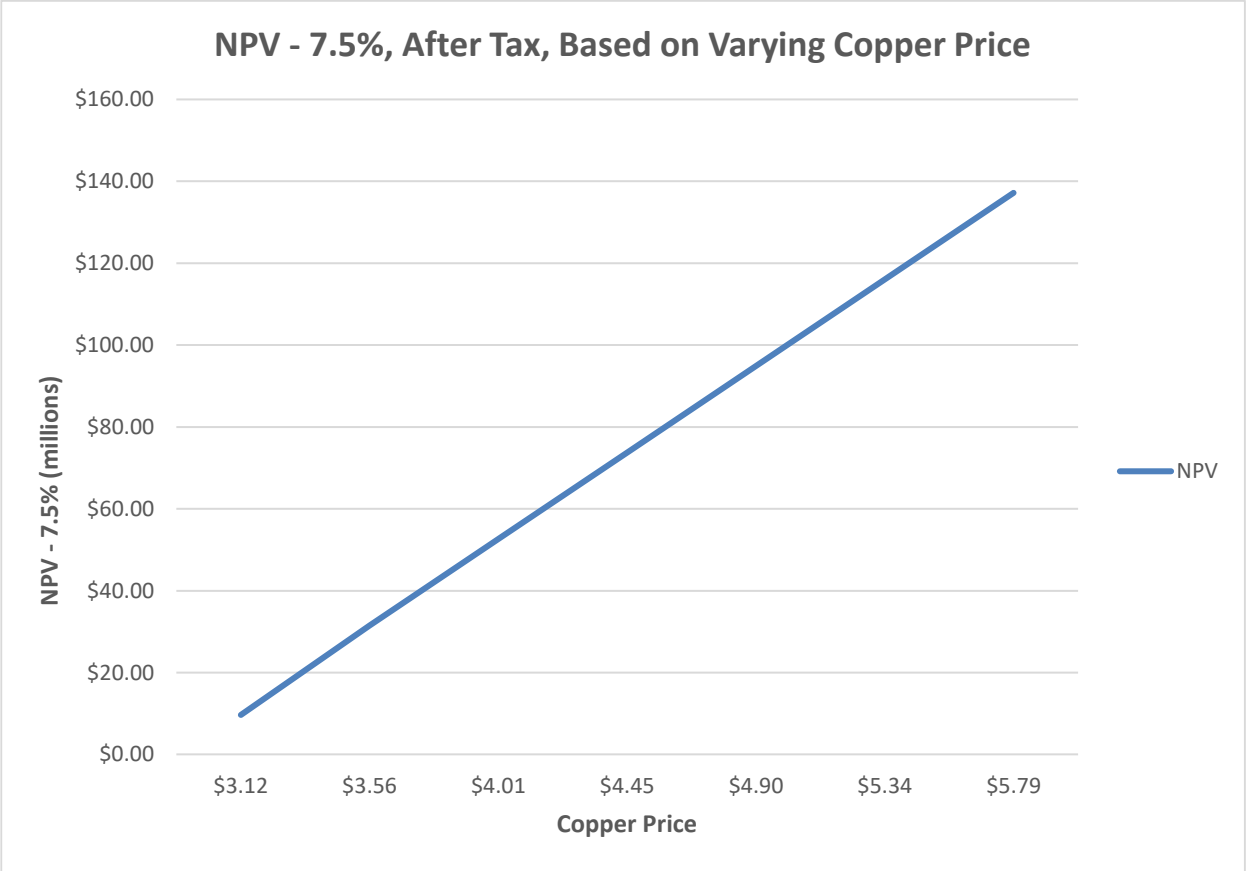


Figure 22-4: Project Copper Price Sensitivity Analysis

22.8.2 Operating Costs, Capital Costs and Metal Price Sensitivity

As discussed, the Project is most sensitive to a change in copper prices, or all metal prices combined. The Project is also very sensitive to the cost of operations, incurring an approximately 19% decline in the NPV-7.5 for each increase of 10% in the operating costs. The Project is less sensitive to variances in the cost of capital, experiencing about 8% in decline in the NPV-7.5 for each increase of 10% in the capital costs, as shown in Figure 22-5. The Project is least sensitive to the price of gold and silver.

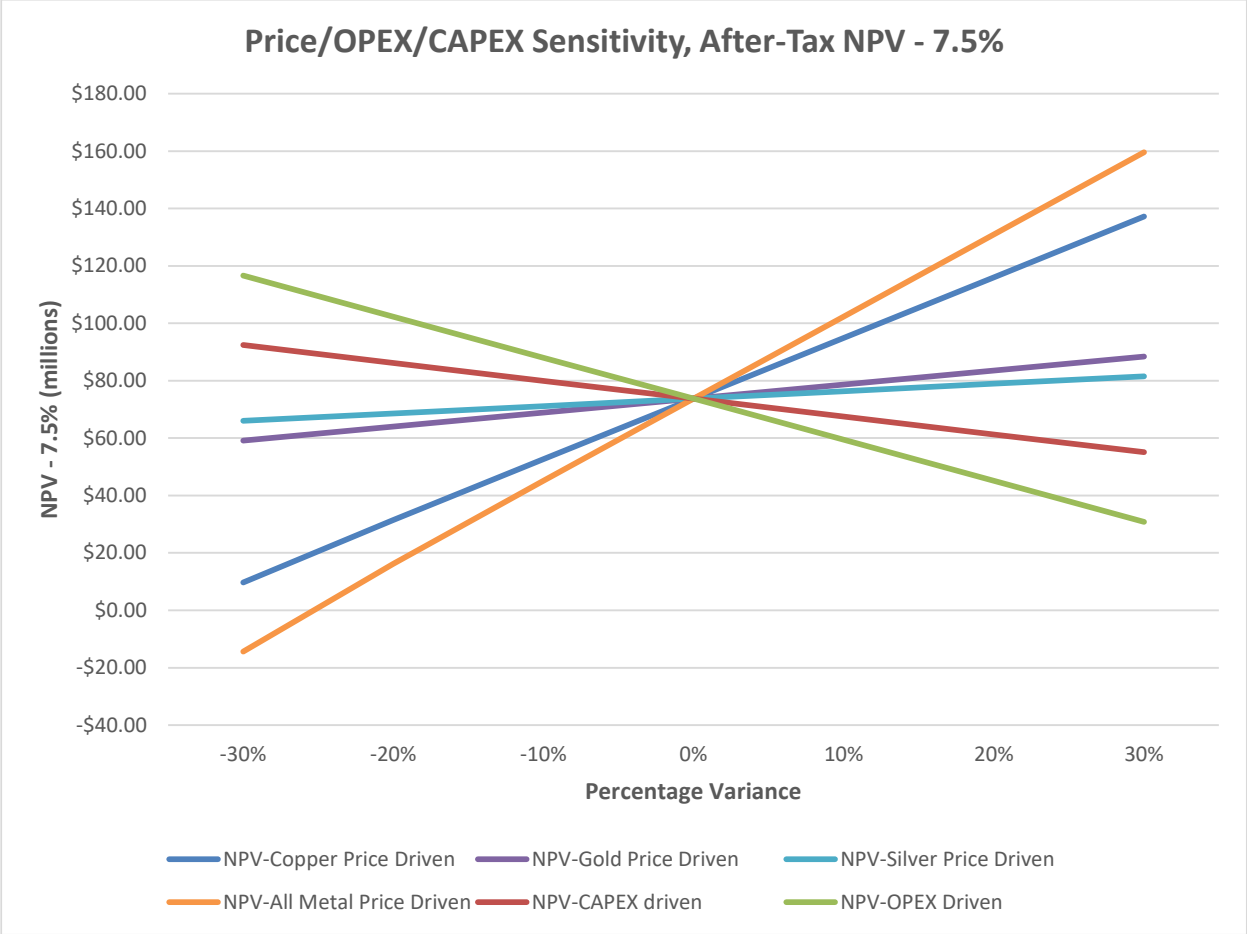


Figure 22-5: Project Operating Cost, Capital Cost and Metal Price Sensitivity Analysis

23. OTHER RELEVANT DATA AND INFORMATION

23.1 Process Plant Geotechnical

The process plant, which consist of crushing, milling, and concentration of metals, was sited with preference to be located near the pit and on private property. The siting also required area for RoM stockpiling. The site located for this study, met that criteria. The Site is located in and adjacent to historic processing area. The area will need to be cleared of legacy infrastructure including foundation prior to installation of proposed facility. The area selected is expected to have shallow or at surface competent rock for supporting the vibrating loads that will be produced by the Process Plant. Further geotechnical investigation is recommended during the next design phase of the project to obtain geotechnical data in the final location of foundations for heavy vibrating loads including the crusher and mill as well as screening structures and concentrator buildings. The investigation is also required to confirm fill material and rock excavation requirements, as well as locating borrow sources.

23.2 Water Management

23.2.1 Water Balance

A mass and water balance model for the concentrator has been developed as an Excel spreadsheet. This model accounts for major uses of water through the entire concentration process. Minor uses, such as dust control, washdown, and reagent solution make-up were not included.

Overall, process water is recycled within the concentrator. There are four streams of process water that leave the concentrator and are not recycled. The most significant outflow is water contained in the tailings that are stacked in the tailings storage facility. These tailings are dewatered to less than 20% moisture before being conveyed to the TSF. Some of that water may be recovered as it drains from the tailings in storage.

The other three streams are the water contained in the flotation concentrate, the water entrained in the precipitated “cement” copper product and finally water in sludge generated by water treatment.

The need for water treatment is driven by an internal imbalance in the concentrator. The acid leach circuit, including leach residue washing, produces more acid water than can be recycled to the start of that circuit. The grinding and flotation circuit has a water deficit due to the water contained in the slurry being delivered to the leach circuit with the flotation tailings. Neutralization of the excess acid water is necessary to allow that water to be recycled to the process water tank to feed the grinding and flotation circuit.

The combined water losses in the four streams listed above amount to approximately 200 gallons per minute assuming that the plant feed is 4000 short tons per day. This will be made up by purchasing water from the City of Mackay.

23.2.2 Water Management Infrastructure

23.2.2.1 Freshwater Supply

The freshwater requirement for the Konnex Empire Mine Project will be approximately 200 gpm. Fresh water will be purchased from the City of Mackay. Fresh water is sourced from a proposed pipe tie in point of existing pipe line from the City Spring to an existing city water tank. The water will be pumped and conveyed through a carbon steel pipeline, following the path of the powerline up to the processing plant. A Pump station will be necessary at the tie in point near the city water tank and one more along the route to the processing plant, with the potential for additional stations depending on the final design.

A carbon steel pipeline will be installed, following the route parallel to the power line and access road for up to the mill site. This pipeline will serve to convey freshwater to the Fire/Fresh Water tank positioned at the mill location. Along the pipeline route, multiple booster stations will be integrated to facilitate the efficient transportation of fresh water to the mill.

23.2.2.2 Potable Water

Potable Water will not be required for the Konnex Empire Mine Project. Portable toilets will be brought to site to provide facilities for employees. Eyewash and safety showers will be self contained units that can be refilled with bottled water from the City of Mackay. Bottled water will be provided onsite for employees.

23.2.2.3 Water Treatment Plant (WTP)

The Water Treatment Plant will treat water utilized in the process for reuse in the process. Test work is ongoing for further design will to be completed in the Feasibility Study.

The primary function of the water treatment plant will be to neutralize the acidic effluent from cementation. The basic unit operations in such a neutralization plant would consist of a lime silo, possibly a slaker, one or more neutralization reactor tanks and a clarifier. The clarifier would use gravity settling to separate a clear, neutralized overflow for recycle and a sludge for disposal. The water treatment plant could be largely automated and monitored by personnel in the concentrator. Treated water will be reused in the process.

23.2.2.4 Storm Water

Stormwater run-on will be diverted away from disturbed areas of the Project. Current layout at or near the top of the ridge will require relatively small diversions and ponds to adequately handle stormwater events. Contact stormwater will be collected in ponds, which may be used for makeup water in the processing facilities project water balance will be prepared in the next level of study to design the pond volumes. It is anticipated that culverts will be required on the access and haul roads where drainages cross to prevent washouts.

The TSF, a lined facility, will feature a sump at the bottom near the embankment. Operational and standby submersible pumps will be installed at the sump's bottom, reclaiming collected water back to the Process Water tank.

23.3 Surface Water Hydrology

The planned mine and mill are located in a watershed on the eastern flank of the White Knob Mountains. Surface water flow is driven by orographic precipitation as snowfall. The annual average precipitation at the mill site (elevation 8,500 feet) is estimated to be 23.13 inches (https://prism.oregonstate.edu/) whereas the 30-year annual precipitation (1981-2010) in the City of Mackay (elevation 5,906 feet) is 9.79 inches (https://wrcc.dri.edu/Climate/west_coop_summaries.php). There are no streams that intersect the proposed areas for development. Surface water drains from the mine and mill area to the south towards Cliff Creek, a perennial stream that flows east to the Big Lost River, and to the north towards an unnamed perennial stream that flows through a culvert on the north side of the Konnex patented claim block before it infiltrates into the ground. A spring with perennial flow of less than 5 gallons per minute (gpm) is located on the north side of the patented claim block on a hillside north of the project that flows a short distance southeast toward the canyon north of the proposed mine (above the historic Alberta Tunnel) before it infiltrates into the ground. Hydrographs from continuous stream flow monitoring since April 2018 on Cliff Creek, at the point of diversion identified in the Konnex water right located approximately one mile southwest of the mill (elevation 8,724 feet), has exhibited sharp annual peaks during spring snowmelt runoff that exceeded 1,000 gpm for less than 1 month before receding to less than 50 gpm during the fall and winter (fourth and first quarters). The City of Mackay holds a water right and diverts water into the city's water distribution system from a spring located approximately 2.5 miles northwest of the mill that discharges 500 to 600 gpm.

23.3.1 Water Balance

A mass balance for the process was developed for this study to determine the water demand, and a source meeting this demand has been identified. Given this and the low PAG of the waste, a site wide water balance was deemed unnecessary at this phase of study. A site wide water balance will be completed in the next phase of study. Project Implementation

23.4 Project Implementation

23.4.1 Project Implementation Strategy

This section outlines a high-level Project Development Strategy, which will be further developed during the next study phase of the Project.

The Technical Report definitions of Scope, Cost and Schedule have been established on the presumption that Konnex will implement the Project utilizing the Engineering, Procurement and Construction Management (EPCM) Execution model. Konnex will appoint an EPCM Contractor (Engineer) with the prerequisite capability and experience to undertake the work.

To complement the EPCM approach, Konnex will adopt Design and Construct (D&C) implementation strategies, for select areas of the Project. Properly executed, the EPCM Execution strategy will afford Konnex the following benefits:

- Lower Capital Cost Outcomes
- Project Implementation flexibility
- Fast-Track Execution opportunities
- Flexible Project Funding Strategies
- Optimal Project Quality Outcomes

23.4.2 Project Organization

Konnex's Project Manager will direct all EPCM and D&C Contractors.

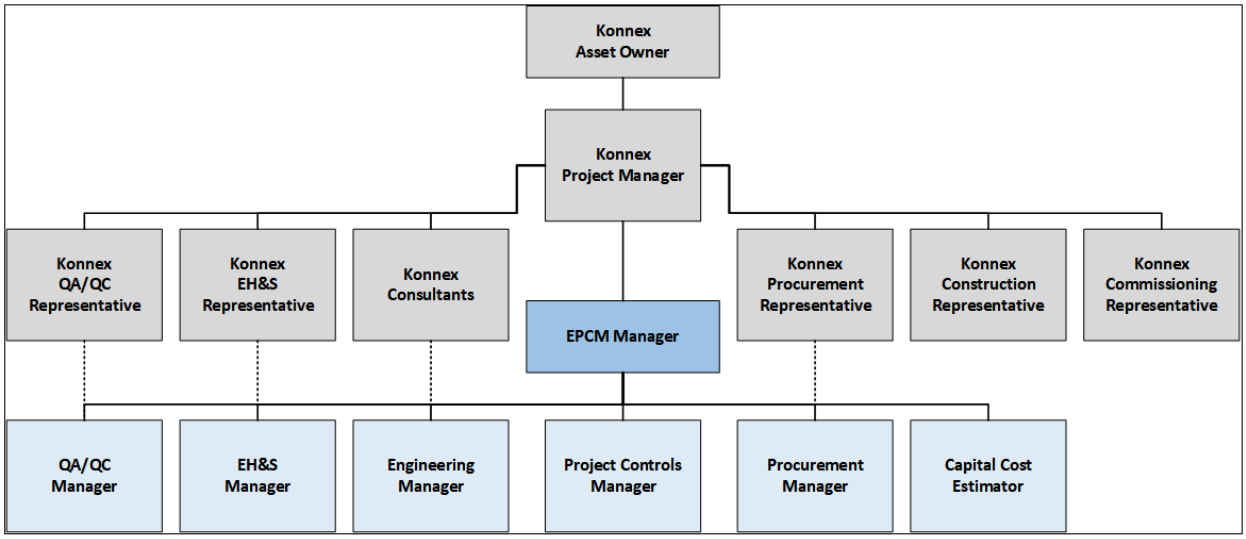


Figure 23-1: Konnex Project Organization Chart

23.4.3 EPCM Management

The Engineer will provide an experienced and suitably qualified Project Manager who will manage all aspects of the EPCM Contract. The EPCM Manager will be the single point of contact for the Empire Project Manager and will work closely with the Konnex senior managers and other Project Managers associated with the project.

23.4.4 Engineering

The Engineer will provide an experienced and suitably qualified Engineering Manager who will manage discipline-based groups of Engineers and Draftsmen that will be responsible for coordination, direction, administration, and completion of all the detail design. Effort will be primarily aimed at optimizing design, uniformity and quality of design and monitoring of time spent against budget.

Where Engineering Design is undertaken, progress will be reported by the Engineering Manager through the EPCM Project Manager to the Empire Project Manager.

23.4.5 EPCM Controls

Utilizing the Feasibility Study report as the basis for project scope and the capital cost estimate as the budget for the beginning of detailed engineering, in accordance with the final Project Schedule that will be submitted in the report.

Initial EPCM activities will include awarding Construction Contracts and/or Supply Contracts for longer lead time items of plant equipment upon Konnex's approval to proceed. The budget and schedule will be continually updated and shared with Konnex to reflect the current understanding of the project cost and status.

- The EPCM Project Controls group will report directly to the EPCM Project Manager and will have the following responsibilities:
- Monitoring and reporting of contract package progress. This will be performed on a daily basis as required and reported weekly and monthly.

- Maintaining and forecasting the cost estimate. Records variations to the budget and other forecast estimate of cost to completion will be updated as necessary and reported.
- Cost control for procurement and contracting. Actual costs (invoiced and payments made) committed costs (orders placed), and estimated costs will be reported against budget. The Trend Notice/Scope Change Notice System will be incorporated with these activities to ensure accurate forecasting.
- Coordination of construction planning and scheduling. Weekly meetings of all site contractors' Project Managers will be held to coordinate changes and priorities between contractors.
- Management of an overall Schedule. The Schedule will be formatted using the WBS information received from all contractors and will be updated using information obtained from the various contractors and reviewed by the Project Manager on a weekly basis as a minimum.
- Project reporting. A monthly project progress report will be developed which will include the following:
    - Safety, Health, and Environmental issues.
    - Project Status.
    - Engineering Progress.
    - Procurement and fabrication progress.
    - Construction activities.
    - Planned activities in the upcoming reporting period.
    - Current project cost reports.
    - Outstanding items.
    - Project S-curves.
    - Photographs depicting progress.

23.4.6 Procurement

The key procurement aims and objectives are to:

- Achieve the project objectives at the earliest possible completion, cost-effective execution, quality workmanship and high degree of safety from suppliers.
- Adhere to the project plan, aims and schedule.
- Ensure that commercial and schedule risks are at acceptable levels.
- Provide a purchasing environment that minimizes claims and protracted disputes.
- Provide a procurement arrangement that encourages suppliers to be innovative and efficient.
- Carry out the procurement function for the project in an ethical and professional manner.

Key success factors are to:

- Meet or exceed expectations for health and safety requirements.
- Meet or exceed the project schedule.
- Meet or exceed the project budget.
- Meet project quality objectives.

23.4.7 Construction Management

The Empire Project's Owner's Team and the EPCM Contractor will oversee all construction activities for the Empire Mine Project. The Konnex team leader will be the Construction and Commissioning Manager and will be supported by a team of construction superintendents who will be directly responsible for oversight for all construction activities related to their discipline. The project superintendents will interface directly with the EPCM Contractor to facilitate the construction management team.

The construction management team will include specialized personnel to direct/oversee the construction and trade contractor quality control and assurance and to provide technical support for all construction. The construction management team will also be accountable for constructability reviews, pre-construction planning, and assistance in construction contracts formation during the project engineering phase.

Upon approval from Konnex the EPCM will provide plans and procedures, or updates to the existing plans and procedures for the following:

- Site boundaries.
- Temporary construction facilities.
- Security.
- Field engineering.
- Site quality management.
- Site logistics.
- Site laydown areas.
- Punch lists, acceptance, and handover.

23.4.8 Commissioning

The Engineer will develop a Commissioning Management Plan, in collaboration with the Konnex Commissioning Representative. The Engineer's Commissioning Manager will report to the EPCM Manager but liaise closely with Konnex's Commissioning Representative. Four main Commissioning Areas have been identified:

- Primary Crusher up to Mill;
- Flotation to Vacuum Drum Filter;
- Leach Circuit Thickener to Cementation Tanks;

- Non-Process Infrastructure.

Supervision of the various areas and disciplines during the discreet commissioning phases will be the responsibility of specifically appointed professional engineers assisted by key personnel from any design teams, construction teams, representatives of the various vendors and from the client's staff.

Commissioning for the Process Plant will be generally carried out in three distinct phases:

- Dry commissioning of all mechanical and electrical equipment including manual rotational checks, off load driven rotational checks, functional checks, instrument I/O checks, electrical continuity checks, etc.;
- Wet commissioning of all mechanical and electrical equipment including hydraulic pressure testing using water, coupled with flow testing using water to ensure integrity of the various pumped circuits; and
- Process commissioning of all mechanical and electrical equipment using production materials, commencing at minimum throughput requirement, and gradually increasing to full design capacity prior to conducting any necessary performance testing.

23.4.9 Temporary Construction Facilities

All Contractors will be responsible for the provision of their own site facilities to an appropriate standard that complies with local EHS guidelines for offices and amenities and to the approval of the Construction Manager.

All contractors will be responsible for the upkeep, cleaning, and sanitation requirements of their respective facilities.

The EPCM Contractor will be responsible for the provision of suitable connection points for power, water, and sewerage. The EPCM Contractor will be responsible for the provision of suitably located areas for the installation of the temporary facilities and for the provision of a suitably located receipt and lay-down area for delivered goods.

23.4.10 Industrial Relations

All contractors will be required, under the terms of their contract, to take responsibility for their own industrial relations. They must be able to demonstrate and have in place suitable policies and procedures to ensure that the handling of matters of an Industrial Relations (IR) nature cause minimum disruption to the project schedule and budget.

All contractors must be able to demonstrate compliance with the HR/IR Policy. The Plan will be incorporated into all tender documentation. This plan will also contain details of the Site Agreement on wages and conditions that will apply universally to the project.

23.4.11 Health and Safety

All contractors must be able to demonstrate compliance with the Environmental, Health, and Safety Project Management Plan. The Plan will be developed during the EPCM phase by the Design and Management Team and incorporated into all tender documentation.

The EPCM Team will employ an experienced Safety Manager for the term of the project and a Site Safety Officer for the period of site occupation.

Contractors will be required to support the project safety protocols, provide individual safety management plans, perform Job Safety Analysis and ensure their employees are provided with Personal Protective Equipment to the standard defined by the overall site policy. Contractors must also provide a nominated individual at supervisory level, who has received adequate training for their representative to be onsite for the period of site occupation.

23.4.12 Environment

The EPCM will ensure that adequate records are kept of all safety incidents, irrespective of whether First Aid is required. TTP's Site Safety Officer will report Lost Time Injury Frequency Rate, Disabling Injury Frequency Rate and Medically Treated Injury Frequency Rate, along with severity information, on a weekly basis as a minimum.

23.4.13 Schedule

23.4.13.1 Schedule Objectives and Scope

The key objective of the PFS phase EPCM schedule is to provide a Class 3, Level 3 detail schedule.

Class of schedule defines the degree completeness required for the development of the schedule with Class 5 being a low degree of completeness, and level 1 being a high degree of completeness.

23.4.13.2 Schedule Assumptions

For the Project, the following assumptions were made for the schedule presented in Table 23-1.

Critical path items are permitting and construction. Procurement and Engineering were pushed out to prevent those items becoming critical path as well;

- The construction schedule may be impacted in the spring months due to snow and/or rain;
- No force majeure disruptions to scheduled work (IR or otherwise);
- Open access to all work fronts is available;
- Transportation to and from site (both land and air) is without delay;
- Work weeks were based on 5 days per week, throughout the schedule.
- The schedule has assumed that project approval will be given by Konnex on or about April 1, 2024. Start up, as defined by handover after completion of commissioning is scheduled early January of 2029.

23.4.13.3 Critical Activities

The Critical Path of the EPCM schedule runs through the Konnex approval process in which permitting, and construction will be priority over that of procurement and engineering. Permitting will be approximately a two-year long process and construction will take place across a year and a half duration.

23.4.13.4 Significant Activities

Major procurement packages for new equipment with a lead time beyond 52 weeks are provided in the table below:

Table 23-1: Items for Procurement

Package	Lead Time
5005 – Agitators	60 Weeks
5006 – Tanks	60 – 80 Weeks
5011-Substation	60-65 Weeks

The Ball Mill has already been procured and will be stored at Empire Project's offsite location in City of Mackay well in advance of execution of this project.

23.4.13.5 Commissioning Schedule

The Commissioning Schedule has been broken into five specific activities in each area / sub area:

- Construction Verification (CV)
  - Occurs immediately after construction completion for each area with each area CV start date independent of the others.
- Pre-Commissioning
  - Occurs once CV is finished for each sub area
  - 10,000t of ore available before pre-commissioning commences
- Dry Commissioning (DC)
  - Requires equipment power up so site power must be completed prior to this step.
  - Should also occur in the order of:
    - Safety Systems (Fire Water / Safety Showers, etc.)
    - Concentrate Ancillary Equipment (Instrument Air / Gland Water)
    - Concentrate equipment substation groupings
    - Check Spares receipted into site store for equipment items in the area
- Wet Commissioning (WC)
  - The order for WC needs to be:
    - Safety Systems (Fire Water / Safety Showers, etc.)
    - Environmental Systems (Storm Water Pond Pumps, Sump Pumps, etc.)
    - Concentrate Ancillary Equipment (Instrument Air / Gland Water)
    - Concentrate Area (Where both the up stream and down stream commissioning has been completed)
    - Workforce training completed

- Process Commissioning
    - Will occur sequentially in the order of process flows, with the provision that each area within a process zone terminated by a large storage buffer has been completed. Large storage buffers likely to create independent process commissioning zones include:
        - Crushed Ore Stockpile
        - Thickener
        - Leach Tanks

23.4.13.6 Schedule Interfaces

The EPCM Schedule does not include detailed activities from contractors undertaking scopes of work outside the PFS scope such as, Mine Development, and LREC work scope from existing substation to the golf course.

The Construction schedule is currently based on the best estimate for the logical sequence of activities as developed by the PFS contractor. Upon award of contracts during the EP phase, construction contractors will be required to submit their own schedules (Class 3, Level 4). This schedule will only be baselined with the approval of Konnex, EPCM contractor, and the construction contractor.

23.4.13.7 Reporting

The Engineer's Project Manager will ensure that the schedule is updated within 3 working days of the end of each calendar month such that progress against project milestones and activities can be clearly identified. The project schedule will also show the critical path at each update such that possible improvements in project completion forecast may be made.

Each month the EPCM contractor will provide the following to Konnex and the contractors:

- The whole schedule
- Critical path / 20 day or less total float view that will identify the critical path while also showing the activities with less than 20 days Total Float

Mid-month short form status report covering expenditure and schedule compliance.

ID	Task Name	Duration	Start	Finish
0	Konnex Execution Schedule - New Equipment	680 days	Jul 1 '24	Feb 5 '27
1	**1 Project Major Milestones**	**672 days**	**Jul 1 '24**	**Jan 26 '27**
2	1.1 Project Approval	1 day	Jul 1 '24	Jul 1 '24
3	1.2 FS Study Complete	1 day	Nov 26 '24	Nov 26 '24
4	1.3 MPO Permitting Complete	1 day	Nov 27 '25	Nov 27 '25
5	1.4 Procurement Begins	1 day	Nov 26 '24	Nov 26 '24
6	1.5 Detailed Design Complete	1 day	Nov 27 '25	Nov 27 '25
7	1.6 Demolition Work Complete	1 day	Mar 16 '26	Mar 16 '26
8	1.7 Earth Works Start	1 day	Mar 16 '26	Mar 16 '26
9	1.8 Concentrate Mill Construction Start	1 day	Jun 22 '26	Jun 22 '26
10	1.9 Dry Commissioning Start	1 day	Nov 30 '26	Nov 30 '26
11	1.10 Construction Completion	1 day	Nov 23 '26	Nov 23 '26
12	1.11 Demobilization	1 day	Nov 23 '26	Nov 23 '26
13	1.12 Produce Concentrate	1 day	Jan 26 '27	Jan 26 '27
14	**2 Project Management**	**367 days**	**Jul 2 '24**	**Nov 26 '25**
15	**2.1 FS Study**	**105 days**	**Jul 2 '24**	**Nov 25 '24**
16	2.1.1 Geotech Field Investigation	30 days	Jul 2 '24	Aug 12 '24
17	2.1.2 Met Testing	50 days	Jul 2 '24	Sep 9 '24
18	2.1.3 Tendor Process	15 days	Sep 10 '24	Sep 30 '24
19	2.1.4 Vendor Reviews	10 days	Sep 10 '24	Sep 23 '24
20	2.1.5 Economics	45 days	Sep 24 '24	Nov 25 '24
21	2.2 MPO Permitting	262 days	Nov 26 '24	Nov 26 '25
22	**3 EPCM**	**518 days**	**Nov 27 '24**	**Nov 20 '26**
23	**3.1 Detailed Design**	**252 days**	**Dec 10 '24**	**Nov 26 '25**
32	**3.2 Procurement**	**339 days**	**Nov 27 '24**	**Mar 16 '26**
33	3.2.1 Procurement Reviews & Awards	79 days	Nov 27 '24	Mar 17 '25
34	**3.2.2 Manufacturing & Shipping**	**260 days**	**Mar 18 '25**	**Mar 16 '26**
35	3.2.2.1 Procurement - Long Lead Items (52+ Weeks)	260 days	Mar 18 '25	Mar 16 '26
36	3.2.2.2 Procurement - Medium Lead Items (26-52 Weeks)	200 days	Mar 18 '25	Dec 22 '25
37	3.2.2.3 Procurement - Short Lead Items (< 26 Weeks)	131 days	Mar 18 '25	Sep 16 '25
38	3.2.2.4 Utilities	131 days	Mar 18 '25	Sep 16 '25
39	**3.3 Construction**	**190 days**	**Mar 2 '26**	**Nov 20 '26**
40	3.3.1 Utilities	33 days	Mar 2 '26	Apr 15 '26
41	3.3.2 Tailings Facility	100 days	Mar 2 '26	Jul 17 '26
42	3.3.3 Roads, Drainage, and Fencing	60 days	Mar 2 '26	May 22 '26
43	3.3.4 Demolition Works	10 days	Mar 2 '26	Mar 13 '26
44	3.3.5 Facility Site Prep	40 days	Mar 16 '26	May 8 '26
45	3.3.6 Concrete Supply	15 days	May 11 '26	May 29 '26
46	3.3.7 Concrete Works	15 days	Jun 1 '26	Jun 19 '26
47	3.3.8 Structural, Mechanical, Piping Install	60 days	Jun 22 '26	Sep 11 '26
48	3.3.9 Field Tankage	10 days	Sep 14 '26	Sep 25 '26
49	3.3.10 Electrical and Instrumentation Install	10 days	Sep 28 '26	Oct 9 '26
50	3.3.11 PLC & SCADA	10 days	Oct 12 '26	Oct 23 '26
51	3.3.12 Communications	20 days	Oct 26 '26	Nov 20 '26
52	3.3.13 Fuel Farm	10 days	May 11 '26	May 22 '26
53	3.3.14 ANFO Facility	30 days	May 11 '26	Jun 19 '26
54	3.3.15 Fire Systems	10 days	Oct 19 '26	Oct 30 '26
55	3.3.16 Site Erection of Buildings	25 days	Sep 14 '26	Oct 16 '26
56	4 Commissioning	45 days	Nov 23 '26	Jan 22 '27
57	4.1 Site Commissioning	45 days	Nov 23 '26	Jan 22 '27
58	4.1.1 Construction Verification	5 days	Nov 23 '26	Nov 27 '26
59	4.1.2 Pre-Commissioning	10 days	Nov 30 '26	Dec 11 '26
60	4.1.3 Dry Commissioning	10 days	Dec 14 '26	Dec 25 '26
61	4.1.4 Wet Commissioning	10 days	Dec 28 '26	Jan 8 '27
62	4.1.5 Process Commissioning	10 days	Jan 11 '27	Jan 22 '27
63	**5 Project Closeout**	10 days	Jan 25 '27	Feb 5 '27
64	**6 Concentrate Production**	1 day	Jan 25 '27	Jan 25 '27

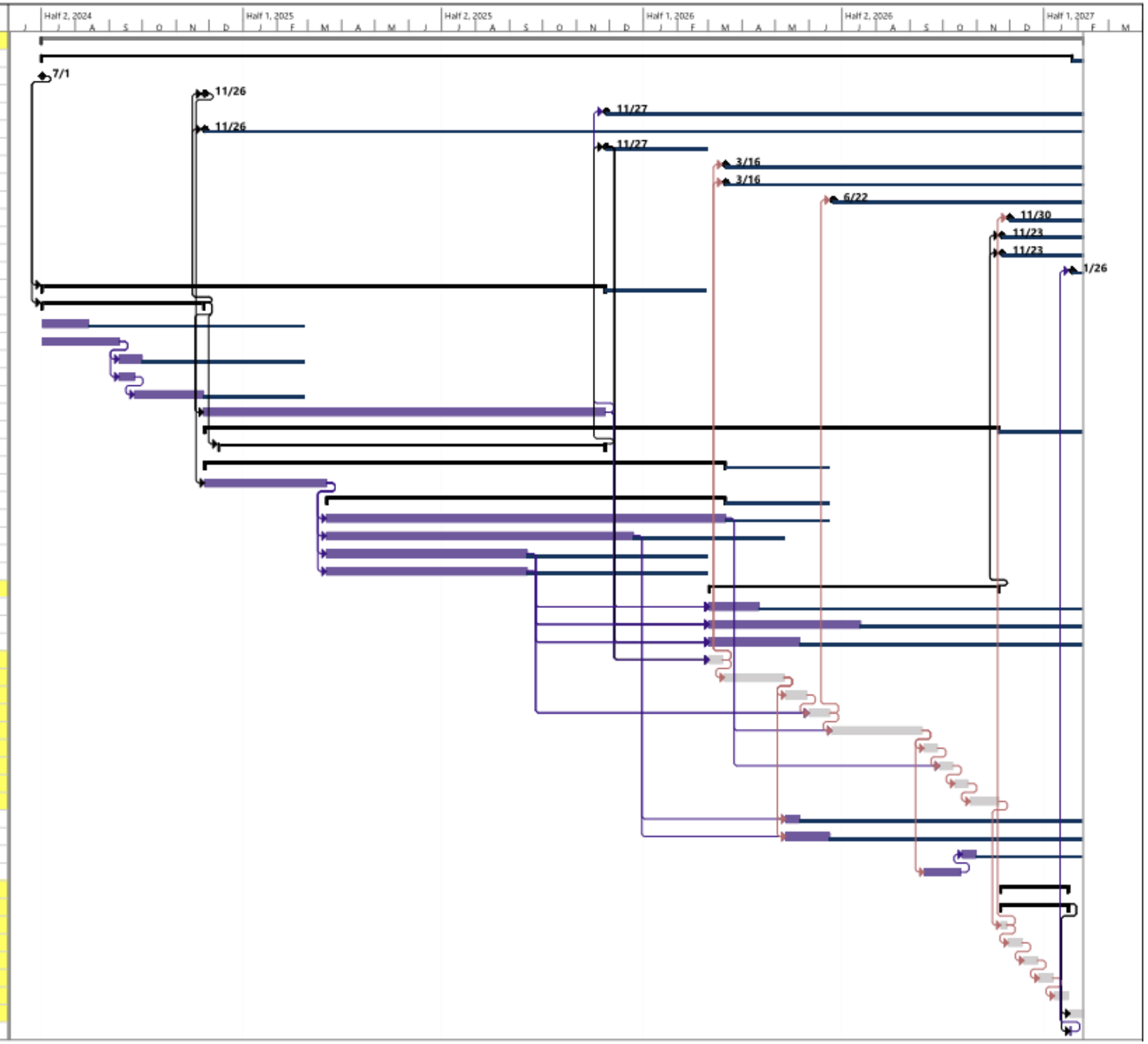


Figure 23-2: Project EPCM Schedule

24. INTERPRETATION AND CONCLUSIONS

24.1 Project Risks

There are several risks in the development of the Empire Mine Project. The risks have been ranked 1 as low, 3 as average, and 5 as high. The mitigation actions during the FS should be undertaken to reduce the overall project risk.

The Table 24-1 lists the risks based on the PFS study.

Table 24-1: Risks Associated with Different Areas

Area	Risk Rank	Description
Geology		
Mining		
Processing	3	Ore Grade being Low
	3	Proportion of Oxide, Sulfide, and Non-Leachable Copper.
	5	Price of Sulfuric Acid
	4	Recovery of Metals
	2	Contracts for saleable copper concentrate and cemented copper.
	1	Penalties in copper concentrate.
Environmental		Handling of Waste, Flotation Tailing, and Beach Residue.
Social	3	Delay in Obtaining Permits

24.2 Geology and Resources

There are no conclusions at this phase of the project.

24.3 Mineral Reserve and Mine Planning

There are no conclusions at this phase of the project.

24.4 Mineral Processing

The amount of metallurgical test work data developed from the Empire mine drill core samples has fed to the development of a robust process flowsheet for recovering copper, gold, and silver. Key conclusions from the metallurgy study are:

- A conventional process flowsheet consisting of crushing, grinding, and rougher and cleaner flotation will recover sulfidized coper oxide, sulfide copper and gold and silver in a saleable copper concentrate.
- The flotation tailing will be conditioned with reagents and carbonates are floated in order to minimize acid consumption for leaching of non-floatable copper oxides.
- Additional copper can be recovered by acid leaching followed by cementation.

24.4.1 Site Preparation

Bulk earthworks are designed to minimize the import of fill materials.

24.4.2 Support Buildings

The office building and security building are planned to be transportable, modular buildings.

24.4.3 Access Roads

The access to the Empire mine follows the existing Smelter Avenue from Mackay to the mine site. Road improvements along Smelter Ave from Mackay are not required until Konnex's property boundary.

24.4.4 Heavy Lifts

Heavy cranage is allowed for all lifts greater than 28 short ton.

24.4.5 Bulk Transport

All bulk transport will be weighed.

24.4.6 Communications

Site wide communications and processing plant data transfer is based upon nearby telephone towers.

24.5 Project Services

Communication with the City of Mackay regarding the utilization of water from the city water tank has not taken place. It is assumed that water can be purchased from the City for a nominal fee, hence the operating costs do not account for this expense.

24.6 Environmental and Social Conclusions

The Project is located on a brownfield site where the proposed mining areas are predominantly on existing areas of disturbance. These aspects of the Project reduce the environmental impacts and permitting hurdles when compared to a greenfield project.

Section 20 summarizes the available information on environmental studies conducted to date and the related known environmental issues associated with the Project; Project permitting requirements and status; Project-related social and community impacts, benefits, and community agreements. Additionally, mine closure, restoration, reclamation, and mitigation are discussed, and attendant costs are estimated to a level of detail that supports Project economic and technical viability to the level of a Feasibility Study and the economics detailed herein.

Considering the current regulatory framework, the Project plans to maximize the use of existing infrastructure to limit new disturbance and include environmental design features to promote environmental protection, the ongoing collaboration between Konnex and the regulatory and administrative agencies at Federal, State, and local levels, and the continued stakeholder engagement actions by Konnex in the local communities and at the regional level. It is reasonable to expect that all required permits and authorizations can be obtained for the Project.

24.7 Recommendations Resource and Exploration

There are no recommendations at this phase of the project.

24.8 Mining Risks and Opportunities

A potential for improving overall economics can be achieved by opening an underground operation.

24.9 Mineral Processing

The ongoing test work is directed towards optimization of the process parameters. The study should lead to reduction of reagent usage thereby reducing operating costs as well as improving gold, silver, and copper recoveries thereby enhancing revenue. A potential that could be encountered is not being able to enhance metals recoveries or the feed grade is lower than projected.

24.10 Environmental Studies

Government agencies will need to review environmental baseline studies for adequacy to identify data gaps and any supplemental work that may be necessary.

24.11 Site-Wide Water Balance

A hydrologic baseline study report will be completed that updates the monitoring data reported in 2021 and incorporates groundwater characterization information (geological logging of boreholes, aquifer testing, and geophysics), as well as watershed-scale water balance models, into a conceptual site model for the project. This report can be completed for a budget of approximately $50,000.

24.12 Groundwater Hydrology and Mine Dewatering

No further groundwater hydrologic studies are recommended at this time, based on the relatively shallow depths of the proposed open pit mines and work to date which indicates the mining activities will not intercept any significant groundwater

24.13 Environmental and Social Conclusions

A draft Project Plan will need to be developed for Project permitting. The estimated cost is approximately $200,000. Based upon a review of similar scale hard rock mines in the United States, the preliminary permitting budget is estimated at $4.0 million +/-30% for a complete Environmental Impact Statement (EIS). Konnex may be able to perform an Environmental Assessment (EA) instead of the EIS which is estimated to cost $500,000 ± 30%.

24.14 Tailings Facility Design

The following studies and investigations are recommended to support TSF design for future phases of the project:

Field Program Consisting of:

Geophysical Study of the TSF Footprint, followed by,

Geotechnical drilling campaign,

24.14.1 Followed by laboratory testing and reporting.TSF Construction Schedule

The TSF staging and construction schedule must be developed in further detail during the feasibility study phase of the project to minimize mobilization costs and assure availability of construction material. The waste rock utilized for starter embankment will need to be reassessed for potential variability in mine plans.

24.14.2 Geotechnical Investigation and Assessment

Geotechnical investigations are recommended, including drilling, sampling, and laboratory testing to characterize the geotechnical properties of the surface and subsurface materials. The geotechnical drilling program should include Standard Penetration Testing (SPT) and/or Cone Penetration Testing (CPT). CPT investigation is recommended to support assessment of the raise foundation strength and liquefaction analysis. A program of laboratory geotechnical testing of a sample of representative future tailings is recommended as part of future design work, including index testing (particle size, plasticity, specific gravity), compaction testing, and advanced testing to assess consolidation, strength and permeability after compaction, and unsaturated soil characteristics.

24.14.3 Waste Rock Testing

Additional laboratory testing of the waste rock is recommended, including, but not limited to, proctor compaction, hydraulic conductivity, and shear strength testing. No testing of the run-of-mine waste was conducted for this study, and a representative range of strength parameters would improve predictions of behavior related to slope stability.

24.14.4 Consolidation/Seepage Modeling

The seepage and stability analyses discussed in this report were based on laboratory tests conducted on the in-situ tailings. Large scale consolidation tests should be conducted on bench scale samples of the proposed process tailings to determine hydraulic conductivities as a function of effective stress. The seepage and stability analyses must be updated based on these representative material properties. Additionally, seepage and stability analyses must be conducted assuming that the underdrain system is plugged to assess its influence on the stability of future vertical expansions.

24.14.5 Water Balance

A TSF water balance analysis which includes the return water pond must be conducted to predict the volume of recycled water available for process operations. The results of the water balance analysis will also be used to size the return water system.

24.14.6 TSF Consequence Classification

A formal dam classification assessment is recommended as part of future design work. Under ANCOLD guidelines, the Consequence Category is determined for a potential dam failure and environmental spill. The classification should include a failure modes assessment, a dam break and inundation study, and consideration of the potential for impacts to business, social and environment, and the potential for loss of life. The resulting Consequence Category for the tailing's storages will be used to identify parameters for advancing the design such as the design storm event and the design seismic event.

24.15 Process Operating Costs

The largest single component of the process operating costs is the acid consumption in the oxide copper leaching circuit. At the lowest level from testing of 100 lb/st acid demand, the cost of the acid represents approximately 69% of the operating cost per ton of ore treated. Therefore, reducing the acid consumption in the leaching circuit will have a major beneficial effect on the overall economics of the project.

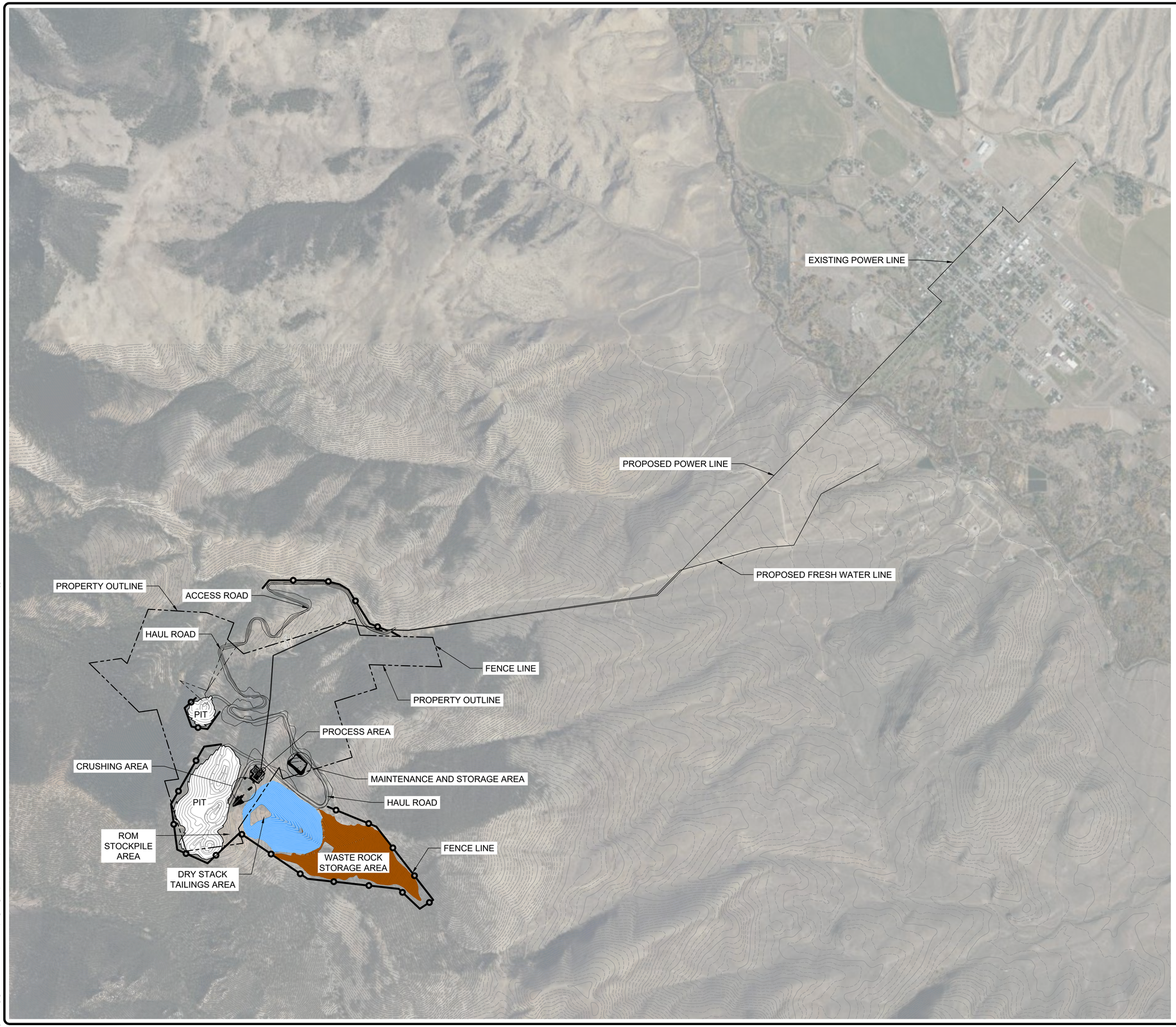
The ore contains variable amounts of limestone (calcium carbonate). Carbon assays of the year-by-year composite samples showed a strong correlation between acid consumption and carbonate carbon content which represents the limestone content. Reducing the amount of limestone in the feed to the leach circuit is possible by adding a limestone flotation stage to the flotation circuit. This will be placed between the sulfide copper rougher flotation stage and the pre-leach thickener, treating the sulfide flotation tailings. Laboratory testing of carbonate flotation is already underway.

25. REFERENCES

1. Sultana Project Metallurgical Results, Cambior USA, March 14, 1997
2. Sultana Project: Acid Cyanide Scoping Consumption Test, Cambior USA, Metcon Research, January 1997
3. Sultana Project: Acid-Cyanide Bottle Roll Leach Study, Metcon Research, February 1997
4. Sultana Project: Acid-Cyanide Bottle Roll Leach Study, Volume III, Metcon Research, April 1997
5. Empire Mine Copper Project Report of Metallurgical Test Work, KCA, August 2005 for Trio Gold Corp
6. Empire Mine Copper Project Report of Metallurgical Test Work, KCA, October 2013 for Boxxer Gold Corp
7. Empire Mine Project Metallurgical Test Work on Leaching Oxide Copper Samples, Mineral Technologies LLC, August 2018
8. Laboratory and Bench-Scale Metallurgical Extraction Tests Performed Using NaCN and ATL on Composite Sample of Magnetite Ore from the Empire Mine, Custer Company, AURIC Met Lab
9. Konnex Resources Empire Mine Acid Leaching Testing, Forte Analytical LLC, May 4, 2022
10. Lithiogeochemistry and Geometallurgy of the Empire Mine Project, Idaho, USA, Phase I Interim Report – DRAFT, March 11, 2022
11. Konnex Resources Inc Empire Mine Ammonium Thiosulfate Test Work Review, March 31, 2022
12. Preliminary Economic Assessment for the Empire Mine Project, Hard Rock Consulting. April 15, 2018.
13. National Instrument 43-101 Technical Report: Updated Mineral Resource Estimate for the Empire Mine Project, Hard Rock Consulting. June 15, 2019.
14. Plan of Operation Report. Konnex Resources, Inc., June 30, 2021.
15. Mineral Processing Equipment Costs and Preliminary Capital Cost Estimations, The Canadian Institute of Mining and Metallurgy. 1978.

APPENDIX A: DRAWINGS

APPENDIX A. 1: GENERAL ARRANGEMENTS



1000 0 1000 2000 3000 Feet

(IN FEET)
1 inch = 1000ft.

ISSUE	DATE	DESCRIPTION			
0	070224	ISSUED FOR USE		SMC	CJG
COORD SYSTEM	AS NOTED		DRAWN	SMC	CJG
			DSGN	CJG	APPD

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE IN WHOLE OR IN PART WITHOUT PERMISSION.

FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE, UNITS 3&4
FORT COLLINS, CO 80524

KONNEX RESOURCES, INC - EMPIRE OPEN PIT PROJECT

PRELIMINARY SITE LAYOUT GENERAL ARRANGEMENT

MACKAY, IDAHO

DRAWING NUMBER:

EXH-0001

FOR INFORMATION ONLY

[Trevon] [Jul 05, 2024 – 2:45pm] [E:\TD_WORK\Jobs\134000 – Konnex Empire Mine\Exhibits\134004 – Konnex_Site-Plan_UT.dwg]

LOST RIVER NEW POWER LINE

KONNEX POWER LINE

EXISTING
TOWN OF MACKAY
WATER TANK

NEW KONNEX WATERLINE
TIE INTO EXISTING

CITY SPRING

FRESH WATER LINE

FENCE LINE

ACCESS ROAD

PROPERTY BOUNDARY

HAUL ROAD

PIT

PROPERTY BOUNDARY

MAINTENANCE PAD AND STORAGE AREA

PROCESS AREA

CRUSHING AREA

HAUL ROAD

PIT

DRY STACK TAILINGS AREA

WASTE ROCK STORAGE AREA

ROM STOCKPILE AREA

FENCE LINE

1000 0 1000 2000 3000 Feet

(IN FEET)
1 inch = 1000ft.

FOR INFORMATION ONLY

ISSUE	DATE	DESCRIPTION					
0	070324	ISSUED FOR USE	TD	SMC	CJG	CJG	
COORD SYSTEM	AS NOTED		DRAWN	DSGN	CHKD	APPD	

THIS DRAWING IS THE PROPERTY OF FORT DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE, IN WHOLE OR IN PART WITHOUT PERMISSION.

FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORT DYNAMICS

120 COMMERCE DRIVE, UNITS 3&4
FORT COLLINS, CO 80524

KONNEX RESOURCES, INC - EMPIRE OPEN PIT PROJECT

PROPOSED UTILITIES
GENERAL ARRANGEMENT

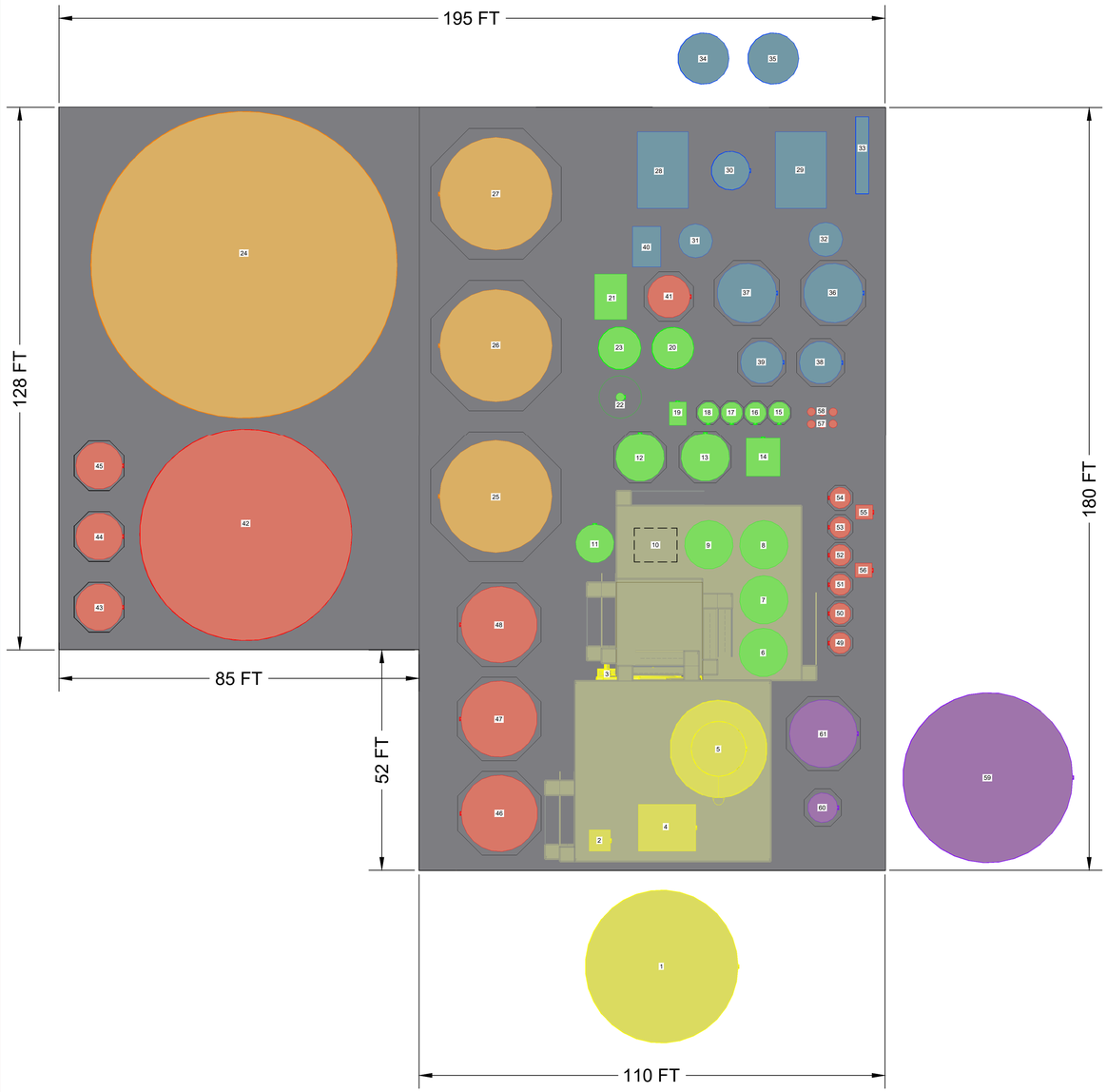
MACKAY, IDAHO

DRAWING NUMBER:

EXH-0002

[Trevon] [Jul 18, 2024 – 4:19pm] [E:\TD_WORK\Jobs\134000 – Konnex Empire Mine\Exhibits\134004 – Konnex_Site-Plan_UT.dwg]

#	Equipment Name	Equipment Number	Major Dimensions
1	Fine Ore Bin	150-BN-001	36' D x 57' H
2	Grinding Media Storage Bin	210-BN-001	5' W x 5' L x 10' H
3	Ball Mill	210-BM-001	15' D x 27' L (total length 47.25')
4	Cyclone Feed Pump Box	210-PB-001	11' W x 13.5' L x 11' H
5	Cyclone Pack	210-CY-001	
6	Rougher Flotation Cell #1	310-FC-001	11.43' D x 12.42' H
7	Rougher Flotation Cell #2	310-FC-002	11.43' D x 12.42' H
8	Rougher Flotation Cell #3	310-FC-003	11.43' D x 12.42' H
9	Rougher Flotation Cell #4	310-FC-004	11.43' D x 12.42' H
10	Rougher Tailings Pump Box	310-PB-001	8' W x 10.1' L x 9' H
11	Carbonate Flotation Conditioning Tank	315-TK-001	9' D x 9' H
12	Carbonate Flotation Cell #1	315-FC-001	11.43' D x 12.42' H
13	Carbonate Flotation Cell #2	315-FC-002	11.43' D x 12.42' H
14	Carbonate Tails Pump Box	315-PB-001	8' W x 9' D x 10.1' H
15	First Stage Cleaner Flotation Cell #1	320-PB-001	5' D x 5' H
16	First Stage Cleaner Flotation Cell #2	320-PB-002	5' D x 5' H
17	First Stage Cleaner Flotation Cell #3	320-PB-003	5' D x 5' H
18	Second Stage Cleaner Flotation Cell #1	325-FC-001	5' D x 5' H
19	Cleaner Tailings Pump Box	320-PB-001	4' W x 5.5' L x 4'
20	Cleaner Concentrate Thickener	330-THK-001	9.8' D x 5.2' H
21	Vacuum Filter Skid	340-FL-001	7.6' X 10.7' L
22	Filtrate Reciever	340-TK-001	2' D x 5' H
23	Vacuum Station	340-VS-001	10' D x 9.2' H
24	Leach Circuit Feed Thickener	410-THK-001	72.2' D X 6.2' H
25	Copper Leach Tank #1	420-TK-001	26.5' D x 28' H
26	Copper Leach Tank #2	420-TK-002	26.5' D x 28' H
27	Copper Leach Tank #3	420-TK-003	26.5' D x 28' H
28	Leach Tailings Disc Filter #1	520-FL-001	12' W X 12' L
29	Leach Tailings Disc Filter #2	520-FL-002	12' W X 12' L
30	Re-Pulp Tank	520-TK-001	9' D X 9' H
31	Vaccum Station #1	520-VS-001	8'D X 5'H
32	Vaccum Station #2	520-VS-002	8'D X 5'H
33	Pug Mill	520-PUG-001	60" W X 11ft L
34	Lime Stone Silo	520-TK-001	12' D x 55.5' H
35	Cement Silo Silo	520-TK-002	12' D x 55.5' H
36	Cementation Surge Tank	530-TK-001	14' D x 14' H
37	Cementation Reactor Tank	530-TK-002	14' D x 14' H
38	Cementation Tank #1	530-TK-003	10' D x 17' H
39	Cementation Tank #2	530-TK-004	10' D x 17' H
40	Cement Copper Wash Screen	530-SR-001	9.5' L x 6.6' W
41	Dilute Acid Storage Tank	630-TK-001	10' D x 10' H
42	Neutralization Clarifier	610-THK-001	50' D
43	Sludge Tank	610-TK-001	11'D X 11'H
44	Iron Oxidation Tank	610-TK-001	11'D X 11'H
45	Nutralization Reactor Tank	610-TK-002	11'D X 11'H
46	Acid Storage Tank #1	630-TK-002	18' D x 18' H
47	Acid Storage Tank #2	630-TK-003	18' D x 18' H
48	Acid Storage Tank #3	630-TK-004	18' D x 18' H
49	Pax Mix Tank	600-TK-002	5' D x 5' H
50	Pax Day Tank	600-TK-003	5' D x 5' H
51	AP404 Mix Tank	600-TK-004	5' D x 5' H
52	AP404 Day Tank	600-TK-005	5' D x 5' H
53	AP208 Mix Tank	600-TK-006	5' D x 5' H
54	AP208 Day Tank	600-TK-007	5' D x 5' H
55	AP208 Tote	-	48" L x 40" W x 53" H
56	AP404 Tote	-	48" L x 40" W x 53" H
57	AF-65 Drum	-	23" D x 34.5" H
58	Fatty Acid Drum	-	23" D x 34.5" H
59	Fire/Fresh Water Tank	710-TK-003	40' D x 40'
60	Seal Water Tank	710-TK-004	7' D x 9' H
61	Process Water Tank	710-TK-001	16' D x 18' H



FOR INFORMATION ONLY

ISSUE	DATE	DESCRIPTION	COORD SYSTEM
0	051424	ISSUED FOR USE	AS NOTED

TD	DRAWN	DSGN	CHKD	CJG	APPD
		SMC	CJG	CJG	

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE, IN WHOLE OR IN PART WITHOUT PERMISSION.

FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE, UNITS 3&4
FORT COLLINS, CO 80524

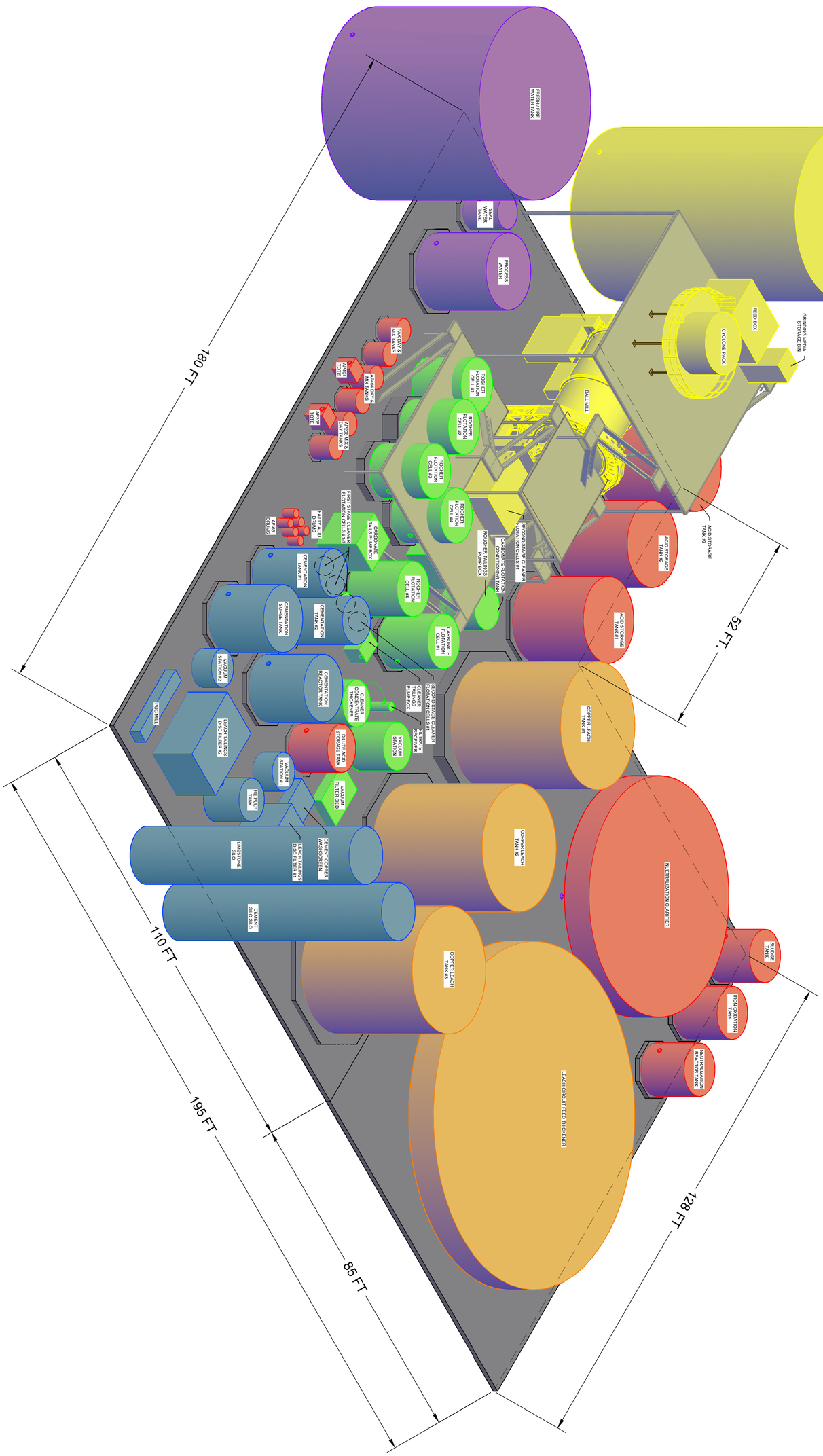
KONNEX RESOURCES, INC - EMPIRE OPEN PIT PROJECT

CONCENTRATOR BUILDING GENERAL ARRANGEMENT

MACKAY, IDAHO

DRAWING NUMBER:
134004-3000-MC

EXH-5001



FOR INFORMATION ONLY

DRAWING NUMBER: 134004-3000-MC

EXH-5002

KONNEX RESOURCES, INC - EMPIRE OPEN PIT PROJECT

CONCENTRATOR BUILDING GENERAL ARRANGEMENT ISO VIEWS

MACKAY, IDAHO

PREPARED BY:

FORTE DYNAMICS

120 COMMERCE DRIVE, UNITS 3&4
FORT COLLINS, CO 80524

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE IN WHOLE OR IN PART WITHOUT PERMISSION.

FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

ISSUE	DATE	DESCRIPTION	DRAWN	DSGN	CHKR	APPD
0	051424	ISSUED FOR USE	TD	SMC	CJG	CJG

COORD SYSTEM: AS NOTED

APPENDIX A. 2: PROCESS FLOW DIAGRAMS

EQUIPMENT/VALVE DESIGNATION

AB	ABSORPTION BED	INS	INLET SILENCER
AC	AIR COMPRESSOR	JBOX	ELECTRICAL JUNCTION BOX
ACD	AIR CONDITION	LOS	LOCAL OPERATOR STATION
AG	AGITATOR	MCC	MOTOR CONTROL CENTER
AN	ANODE	MCS	MOTOR CONTROL SILENCER
ATS	AUTOMATIC TRANSFER SWITCH	MD	MOTORIZED DAMPER
B	BASIN (CONCRETE)	ME	MIST ELIMINATOR
BC	BLOWOFF COLUMN	MS	FUME HOOD
BG	BUILDING	MX	MIXER
BH	BAGHOUSE	NPS	NON-POTABLE WATER PUMP STATION
BS	BAG STATION	OCS	ODOR CONTROL SYSTEM
BL	BLOWER	OTS	OUTLET SILENCER
BM	BALL MILL	PB	PUMP BOX
BN	BIN	PD	POND
BOS	BLOWOFFF SILENCER	PDC	POWER DISTRIBUTION CENTER
CC	CALIBRATION COLUMN	PLC	PROGRAMMABLE LOGIC CONTROLLER
CD	CONTROL DAMPER	PN	PAN
CF	CENTRIFUGE	PP	POWER PANEL
CFS	CHEMICAL FEED PUMP	PO	POT
CI	CHEMICAL POT FEEDER	PUG	PUG MILL
CP	CONTROL PANEL	PS	POWER SUPPLY
CR	CRUSHER	PU	PUMP
CV	CONVEYOR	RA	AIR RECEIVER
CX	CATHODE	RB	ROCK BREAKER
CY	CYCLONE	RM	ROD MILL
DR	DRYER	RS	RADIAL STACKER
DS	DIVERSION STRUCTURE	RT	ROCK TRAP
DSP	DISPENSER	RV	RECEIVER
ED	EDUCTOR	SA	SAMPLER
EM	EMERGENCY EYEWASH	SB	SPLITTER BOX
EN	ENCLOSURE	SCV	SCREW CONVEYOR
ERV	ENERGY RECOVERY VENTILATOR	SDG	SLIDE GATE
EW	ELECTROWINNING CELL	SF	EYEWASH & SAFETY SHOWER
EXF	EXHAUST FAN	SK	SKID
F	FAN	SLG	SLUICE GATE
FA	FIRE ALARM SYSTEM	SM	SUBMERSIBLE MIXER
FE	FEEDER	SM	SPRAY MONITOR
FL	FILTER	SOV	SOLENOID VALVE
FC	FLOTATION CELL	SQ	SEPARATOR
FN	FURNACE	SR	SCREEN
FMS	FUEL MANAGEMENT SYSTEM	SRS	SEPTAGE RECIEVING STATION
GC	GRAVITY CONCENTRATOR	STG	STOP GATE
GDR	GRINDER	SWC	SCREENINGS WASHER/COMPACTOR
GEN	GENERATOR	SWM	SEAL WATER MONITOR
GF	GREASE FILTER	TB	TRASH BIN
GR	GROUNDING ROD	TC	TURBO COMPRESSOR
GRN	GROUNDING RING	THK	THICKENER
HD	HOOD	TK	TANK
HF	HARMONIC FILTER	UVB	UV DISINFECTION BANK
HL	HEAP LEACH PAD	UVPD	UV POWER DISTRIBUTION CENTER
HO	HOIST	UVHC	UV HYDRAULIC SYSTEM CENTER
HOS	HOSE REEL	UWB	UNDERCARRIAGE WASH BAR
HP	HOPPER	VEF	VENTILATION EXHAUST FAN
HSV	HOSE VALVE	VF	VIBRATORY FEEDER
HT	HEATER	VFD	VARIABLE FREQUENCY DRIVE
HWH	HOT WATER HEATER	VSF	VENTILATION SUPPLY FAN
HX	HEAT EXCHANGER	WM	WATER METER
IBP	INSTRUMENTATION BIOREACTOR PANEL	WS	WYE-STRAINER
IDS	INTERNALLY-FED DRUM SCREEN	WT	WATER SOFTENER
INF	INLET FILTER	YH	YARD HYDRANT

GENERAL NOTES

1. THIS IS A STANDARD LEGEND, THEREFORE, NOT ALL OF THIS INFORMATION MAY BE USED ON THIS PROJECT.

EQUIPMENT NUMBERS

XX-YYY-ZZ

- GROUP DESIGNATION
- EQUIPMENT/PROCESS DESIGNATION
- NUMERICAL DESIGNATION

GROUP DESIGNATION

SITE – GENERAL	000	DRY CONCENTRATE BIN	360
CRUSHING – GENERAL	100	ACID LEACHING – GENERAL	400
PRIMARY CRUSHING	110	LEACH CIRCUIT FEED THICKENER	410
SECONDARY CRUSHING	120	COPPER LEACHING CIRCUIT	420
CRUSHING PLANT SCREEN	130	TAILINGS – GENERAL	500
TERTIARY CRUSHING	140	LEACH (FINAL) TAILINGS FILTER	510
FINE ORE BIN	150	LEACH (FINAL) TAILINGS FILTER	520
GRINDING – GENERAL	200	COPPER CEMENTATION	530
MILLING CIRCUIT	210	REAGENTS – GENERAL	600
FLOTATION – GENERAL	300	WATER TREATMENT	610
ROUGHER FLOTATION CIRCUIT	310	LIME	620
CLEANER FLOTATION CIRCUIT	320	ACID	630
CLEANER CONCENTRATE THICKENER	330	WATER	700
CONCENTRATE FILTER	340	UTILITIES – GENERAL	800
CONCENTRATE DRYER	350		

PUMP AND COMPRESSOR SYMBOLS

- CENTRIFUGAL PUMP (DRY PIT)
- CENTRIFUGAL WET PIT PUMP OR TURBINE PUMP
- CHEMICAL FEED PUMP
- GEAR PUMP OR BLOWER (POSITIVE DISPLACEMENT)
- PISTON PUMP
- DIAPHRAGM PUMP
- METERING PUMP
- VERTICAL TURBINE PUMP
- DISCHARGE PUMP
- VACUUM PUMP
- VACUUM PUMP
- SUBMERSIBLE SUMP PUMP
- COMPRESSOR (CENTRIFUGAL), BLOWER OR TURBINE
- COMPRESSOR (PISTON)
- EJECTOR
- SUBMERSIBLE VERTICAL PUMP
- SLIDING VANE PUMP

NOTE: XX:
AS ADJUSTABLE SPEED
CS-1 CONSTANT SPEED (SINGLE SPEED)
CS-2 CONSTANT SPEED (TWO SPEED)

MISCELLANEOUS SYMBOLS

- BLIND FLANGE
- BREATHER
- HOT WATER HEATER
- SELF CONTAINED AIR SUPPLY
- HOSE CONNECTION
- PIPE CAP
- FLEX HOSE
- FILTER
- FLAME ARRESTER
- WARNING LIGHT
- CLOCK GAUGE
- BLOWER
- HOSE REEL RACK
- VENTILATION FAN
- COMPRESSED AIR TANK
- CHUTE
- PFD-
- EQUIP NAME XXXXX-XX-XXX
- REDUCER
- BASKET STRAINER
- WATER SURFACE
- SILENCER
- FLUSHING CONNECTION
- SEAL WATER SET
- PULSATION DAMPENER
- ROCK TRAP
- HEAT EXCHANGER
- SCREENINGS WASHER/COMPACTOR
- REDUCER
- CAP
- ALARM
- EMERGENCY VENT
- JOHNSON FILTER
- AGITATOR
- 000 MASS BALANCE NUMBERS

LINE SYMBOLS

PROCESS LINES

- MAJOR PROCESS
- MINOR PROCESS
- MAJOR EXISTING PROCESS
- MINOR EXISTING PROCESS

ISSUE	DATE	DESCRIPTION	COORD SYSTEM
0	052424	ISSUED FOR USE	AS NOTED

LB	DRAWN		
DM	DSGN		
SMC	CHKD	DM	
	APPD		

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE IN WHOLE OR IN PART WITHOUT PERMISSION.
FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

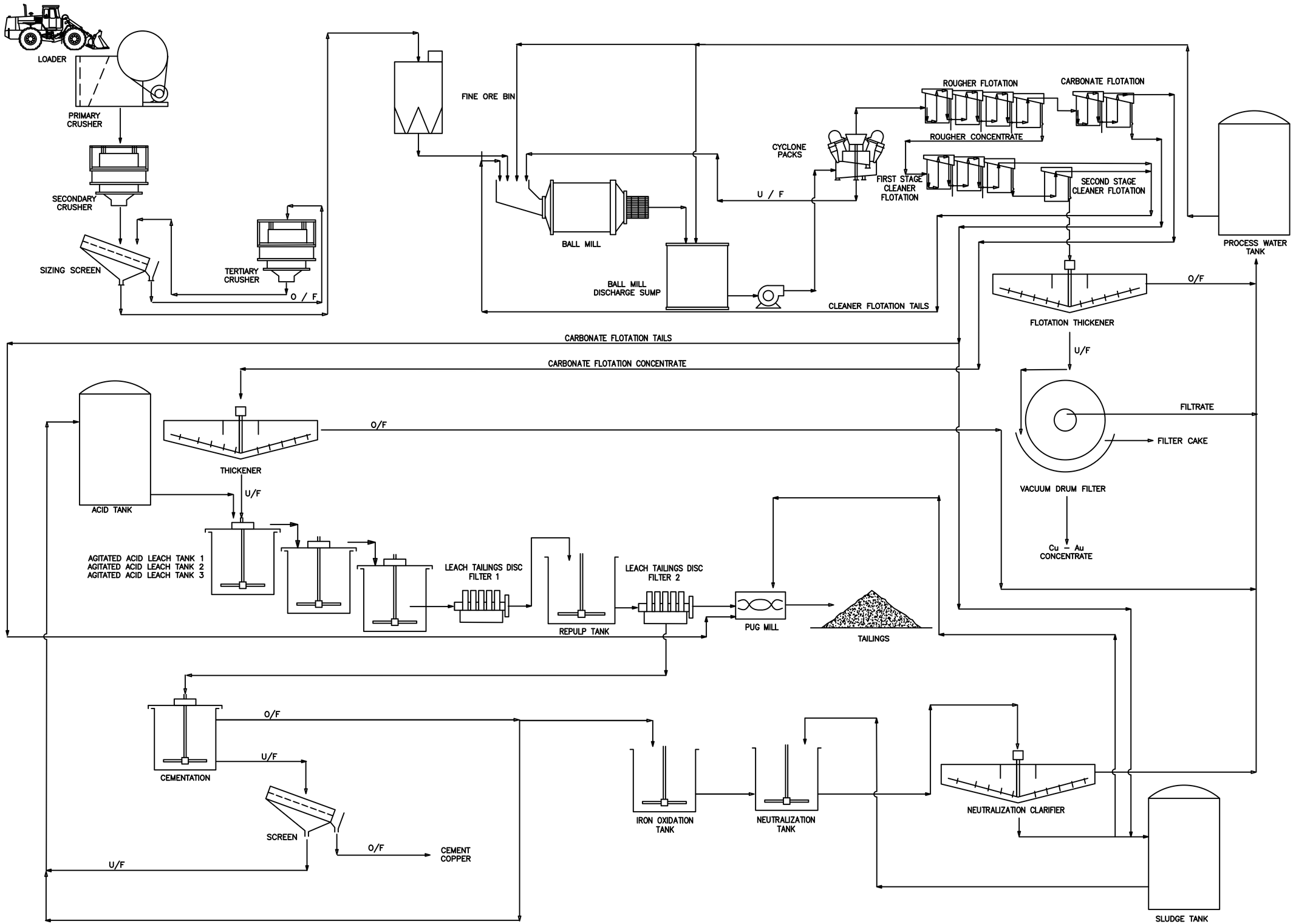
PREPARED BY:
FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

KONNEX EMPIRE MINE

NOTES & LEGEND

MACKAY, IDAHO

DRAWING NUMBER:
PFD-6000
SHEET 1 OF 10



ISSUE	DATE	DESCRIPTION	DRAWN	DSGN	CHKD	APPD
0	052424	ISSUED FOR USE	LB	DM	SMC	DM
		COORD SYSTEM: AS NOTED				

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE IN WHOLE OR IN PART WITHOUT PERMISSION.

FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

KONNEX EMPIRE MINE

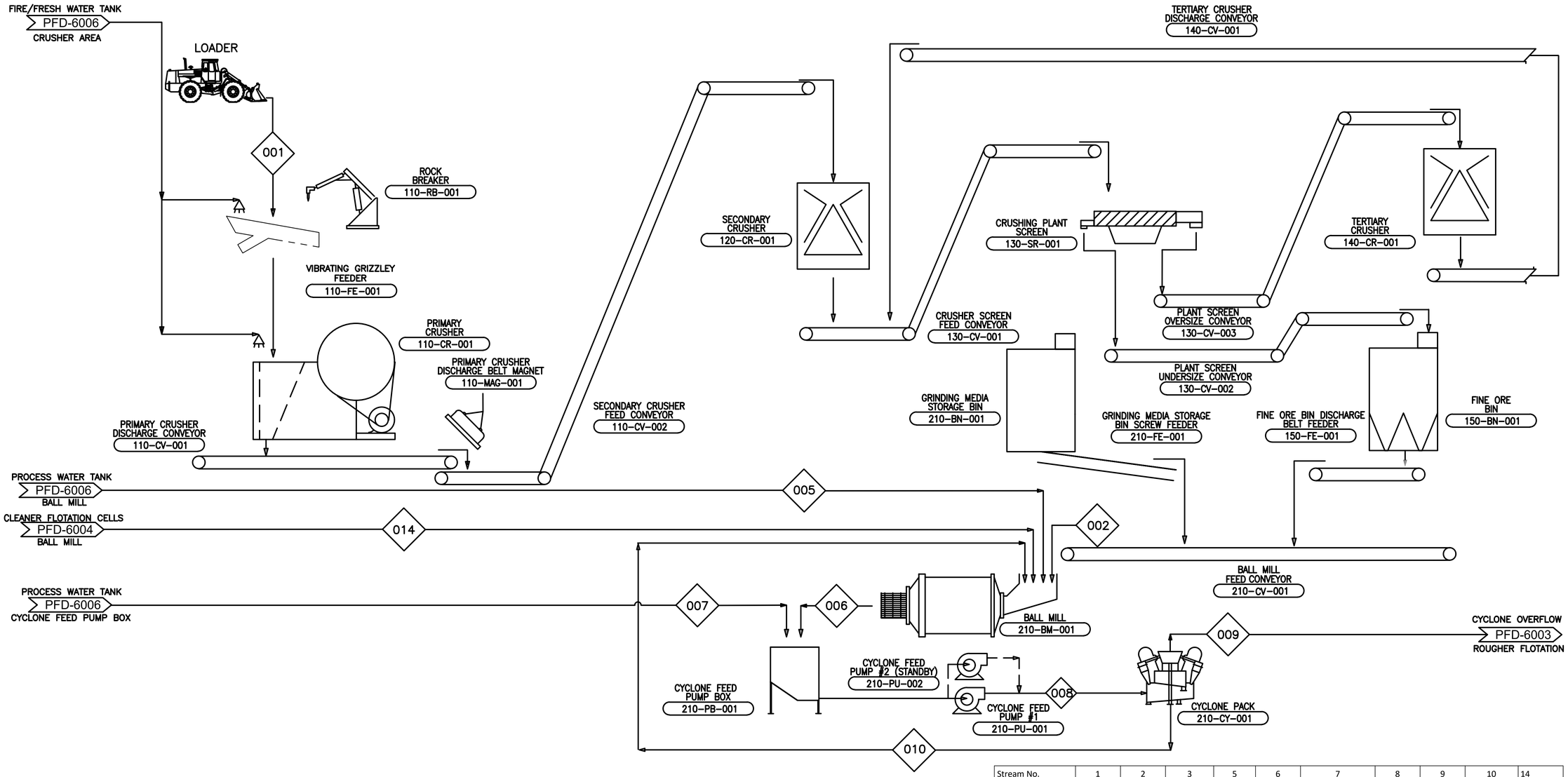
OVERALL FLOW SHEET

MACKAY, IDAHO

DRAWING NUMBER:

PFD-6001

SHEET 2 OF 10



Stream No.	1	2	3	5	6	7	8	9	10	14
Stream Name	Crushing	Grinding New Feed	Grinding Circulating Load	Mill Feed Water Addition	Ball Mill Discharge	Cyclone Feed Pump Box Water Addition	Cyclone Bank Feed	Cyclone Overflow	Cyclone Underflow	Cleaner Flotation Tails
Hours/day	20.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0
Solids st/hr	200.0	166.7	416.7	0.0	588.0	0.0	588.0	166.7	416.7	4.7
% Solids	1.0	1.0	0.7	0.0	0.7	0.0	0.5	0.4	0.7	0.2
Water st/hr	10.0	8.3	224.4	67.1	316.6	217.3	533.9	309.5	224.4	16.8
Total Slurry Mass Flow st/hr	210.0	175.0	641.0	67.1	904.6	217.3	1121.9	476.2	641.0	21.5
Slurry Volume Flow cuft/min	58.7	48.9	231.0	35.9	325.9	116.1	442.0	209.8	231.0	10.2
Slurry Volume Flow gpm	438.9	365.7	1727.6	268.3	2438.0	868.1	3306.1	1569.2	1727.6	76.4
Copper Mass lb/hr	2000.0	1666.7	4166.7	NA	5909.3	NA	5909.3	1666.7	4166.7	29.2
Gold Mass troz/hr	2.9	2.4	6.1	NA	8.6	NA	8.6	2.4	6.1	0.1
Silver Mass troz/hr	109.1	90.9	227.3	NA	322.2	NA	322.2	90.9	227.3	1.5
Copper Grade							0.0	0.0		0.0
Gold Grade troz/st							0.0	0.0		0.0
Silver Grade troz/st							0.5	0.5		0.3

ISSUE	DATE	DESCRIPTION	DRAWN	CHKD	SNC	DM
0	052424	ISSUED FOR USE	LB	DM		APPD
COORD SYSTEM	AS NOTED					

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE IN WHOLE OR IN PART WITHOUT PERMISSION.
FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

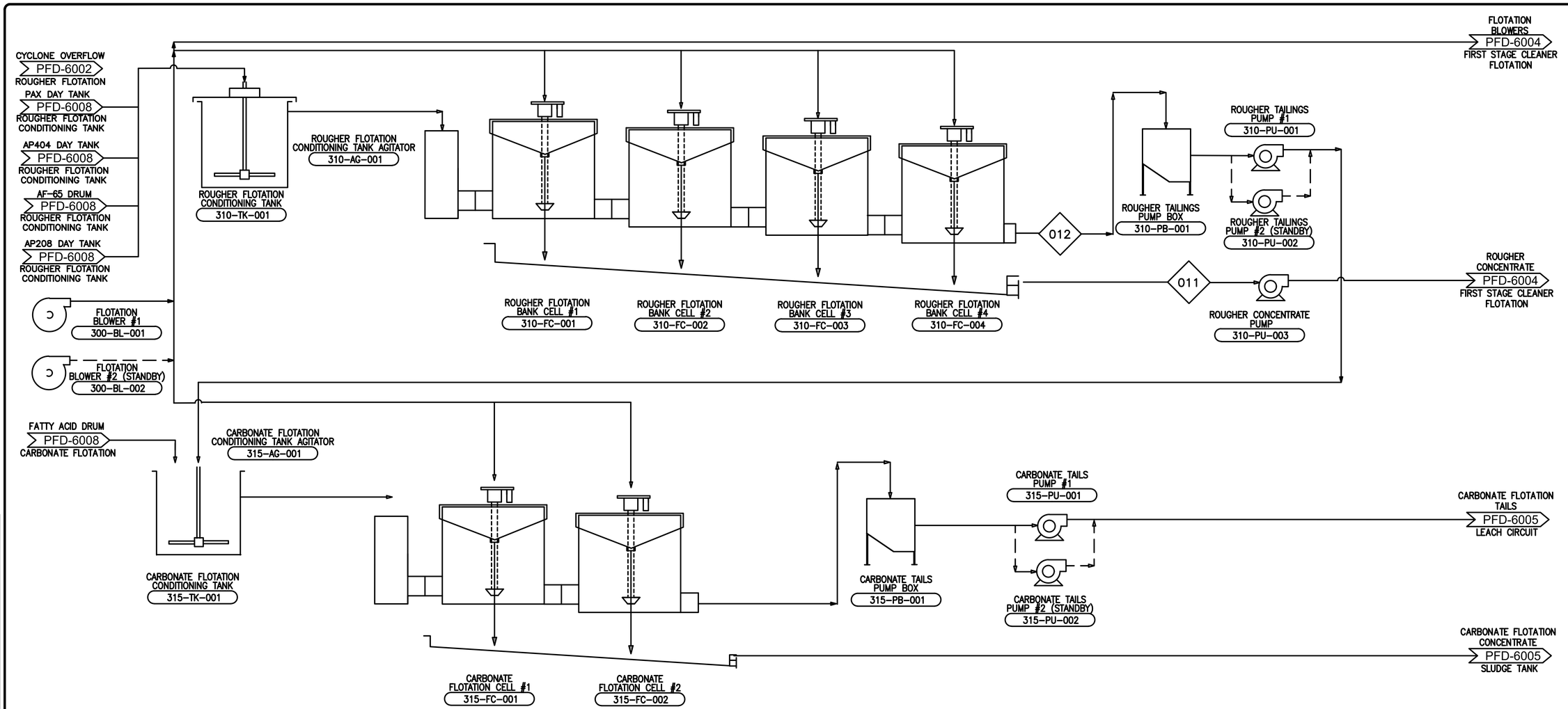
KONNEX EMPIRE MINE

SORTING, CRUSHING, AND GRINDING

MACKAY, IDAHO

DRAWING NUMBER:

PFD-6002



Stream No	11	12
Stream Name	Rougher Flotation Conc.	Rougher Flotation Tails
Hours/day	24.0	24.0
Solids st/hr	5.8	160.8
% Solids	0.2	0.4
Water st/hr	23.3	286.2
Total Slurry Mass Flow st/hr	29.2	447.0
Slurry Volume Flow cuft/min	14.0	195.8
Slurry Volume Flow gpm	104.9	1464.3
Copper Mass lb/hr	583.3	1083.3
Gold Mass troz/hr	1.3	1.2
Silver Mass troz/hr	30.0	60.9
Copper Grade	0.1	0.0
Gold Grade troz/st	0.2	0.0
Silver Grade troz/st	5.1	0.4

ISSUE	DATE	DESCRIPTION	COORD SYSTEM
0	052424	ISSUED FOR USE	AS NOTED

LB	DRAWN		
DM	DSGN		
SMC	CHKD		
DM	APPD		

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE, IN WHOLE OR IN PART WITHOUT PERMISSION.
FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

KONNEX EMPIRE MINE

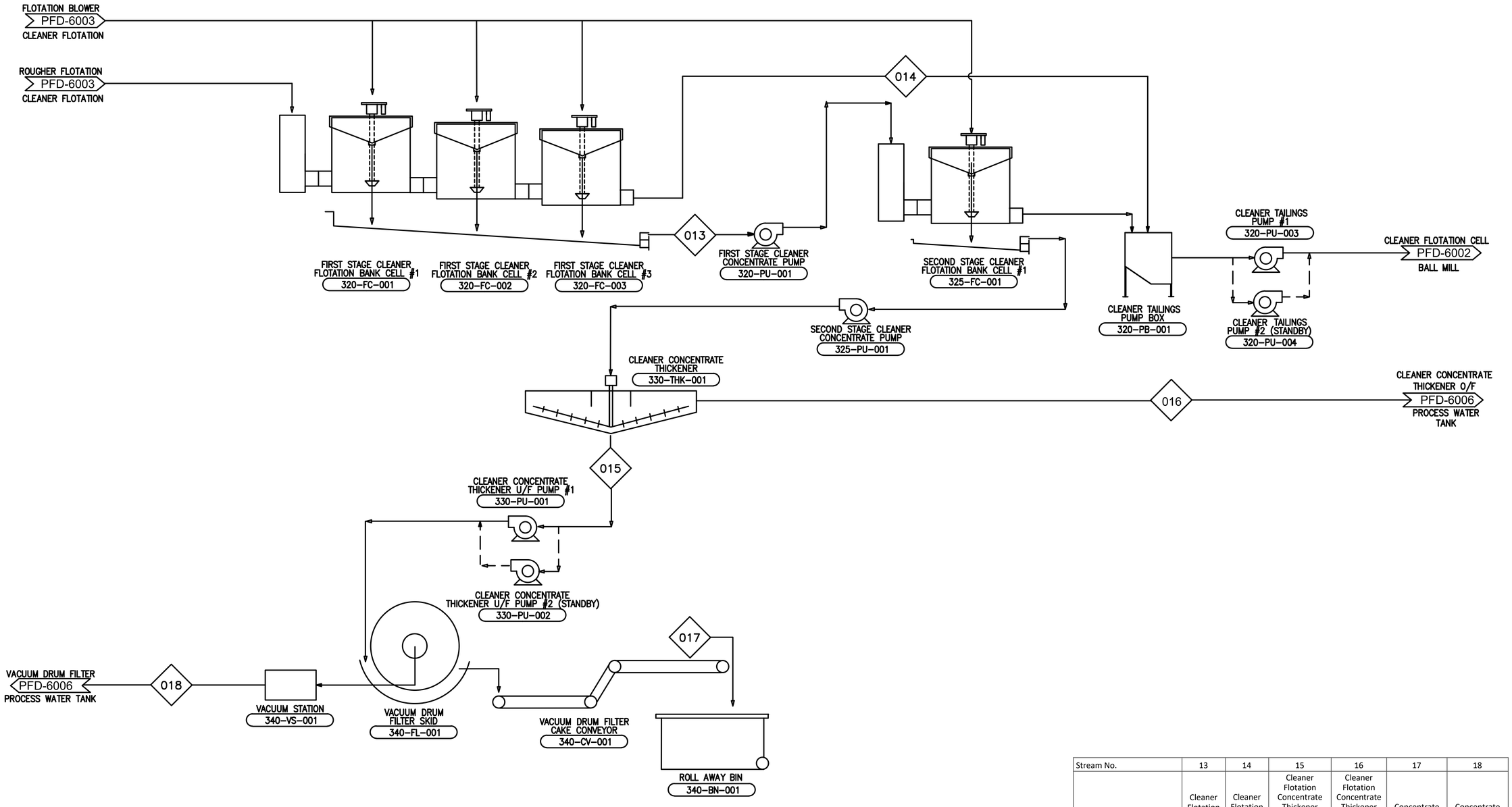
ROUGHER FLOAT AND CARBONATE FLOTATION

MACKAY, IDAHO

DRAWING NUMBER:

PFD-6003

SHEET 4 OF 10



Stream No.	13	14	15	16	17	18
Stream Name	Cleaner Flotation Conc.	Cleaner Flotation Tails	Cleaner Flotation Concentrate Thickener Underflow	Cleaner Flotation Concentrate Thickener Overflow	Concentrate Filter Cake	Concentrate Filtrate
Hours/day	24.0	24.0	24.0	24.0	24.0	24.0
Solids st/hr	1.2	4.7	1.2	0.0	1.2	0.0
% Solids	0.2	0.2	0.7	0.0	0.8	0.0
Water st/hr	6.5	16.8	0.6	5.9	0.3	0.3
Total Slurry Mass Flow st/hr	7.7	21.5	1.8	5.9	1.4	0.3
Slurry Volume Flow cuft/min	3.8	10.2	0.6	3.2	0.5	0.2
Slurry Volume Flow gpm	28.4	76.4	4.8	23.7	3.5	1.3
Copper Mass lb/hr	554.2	29.2	554.2	NA	554.2	NA
Gold Mass troz/hr	1.2	0.1	1.2	NA	1.2	NA
Silver Mass troz/hr	28.5	1.5	28.5	NA	28.5	NA
Copper Grade	0.2	0.0	0.2		0.2	
Gold Grade troz/st	1.0	0.0	1.0		1.0	
Silver Grade troz/st	24.7	0.3	24.7		24.7	

ISSUE	DATE	DESCRIPTION	COORD SYSTEM
0	052424	ISSUED FOR USE	AS NOTED

DRAWN	DSGN	CHKD	APPD
LB	DM	SMC	DM

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE IN WHOLE OR IN PART WITHOUT PERMISSION.
FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

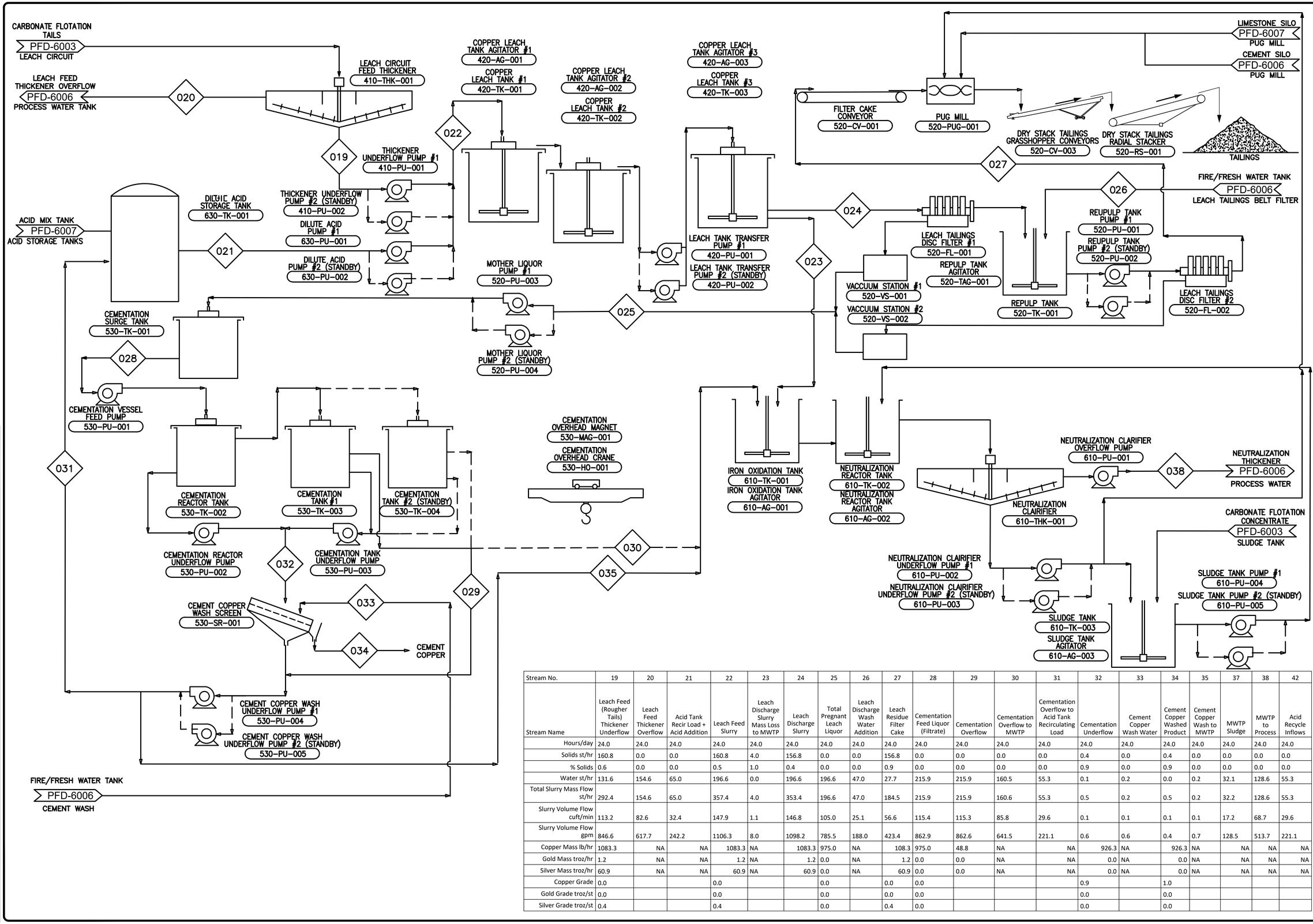
KONNEX EMPIRE MINE

CLEANER FLOTATION AND CLEANER CONCENTRATE FILTRATION

MACKAY, IDAHO

DRAWING NUMBER:

PFD-6004



Stream No.	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	37	38	42
Stream Name	Leach Feed (Rougher Tails) Thickener Underflow	Leach Feed Thickener Overflow	Acid Tank Recir Load + Acid Addition	Leach Feed Slurry	Leach Discharge Slurry Mass Loss to MWTP	Leach Discharge Slurry	Total Pregnant Leach Liquor	Leach Discharge Wash Water Addition	Leach Residue Filter Cake	Cementation Feed Liquor (Filtrate)	Cementation Overflow	Cementation Overflow to MWTP	Cementation Overflow to Acid Tank Recirculating Load	Cementation Underflow	Cement Copper Wash Water	Cement Copper Washed Product	Cement Copper Wash to MWTP	MWTP Sludge	MWTP to Process	Acid Recycle Inflows
Hours/day	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0	24.0
Solids st/hr	160.8	0.0	0.0	160.8	4.0	156.8	0.0	0.0	156.8	0.0	0.0	0.0	0.0	0.4	0.0	0.4	0.0	0.0	0.0	0.0
% Solids	0.6	0.0	0.0	0.5	1.0	0.4	0.0	0.0	0.9	0.0	0.0	0.0	0.0	0.9	0.0	0.9	0.0	0.0	0.0	0.0
Water st/hr	131.6	154.6	65.0	196.6	0.0	196.6	196.6	47.0	27.7	215.9	215.9	160.5	55.3	0.1	0.2	0.0	0.2	32.1	128.6	55.3
Total Slurry Mass Flow st/hr	292.4	154.6	65.0	357.4	4.0	353.4	196.6	47.0	184.5	215.9	215.9	160.6	55.3	0.5	0.2	0.5	0.2	32.2	128.6	55.3
Slurry Volume Flow cuft/min	113.2	82.6	32.4	147.9	1.1	146.8	105.0	25.1	56.6	115.4	115.3	85.8	29.6	0.1	0.1	0.1	0.1	17.2	68.7	29.6
Slurry Volume Flow gpm	846.6	617.7	242.2	1106.3	8.0	1098.2	785.5	188.0	423.4	862.9	862.6	641.5	221.1	0.6	0.6	0.4	0.7	128.5	513.7	221.1
Copper Mass lb/hr	1083.3	NA	NA	1083.3	NA	1083.3	975.0	NA	108.3	975.0	48.8	NA	NA	926.3	NA	926.3	NA	NA	NA	NA
Gold Mass troz/hr	1.2	NA	NA	1.2	NA	1.2	0.0	NA	1.2	0.0	0.0	NA	NA	0.0	NA	0.0	NA	NA	NA	NA
Silver Mass troz/hr	60.9	NA	NA	60.9	NA	60.9	0.0	NA	60.9	0.0	0.0	NA	NA	0.0	NA	0.0	NA	NA	NA	NA
Copper Grade	0.0			0.0			0.0		0.0	0.0				0.9		1.0				
Gold Grade troz/st	0.0			0.0			0.0		0.0	0.0				0.0		0.0				
Silver Grade troz/st	0.4			0.4			0.0		0.4	0.0				0.0		0.0				

ISSUE	DATE	DESCRIPTION
0	052424	ISSUED FOR USE
COORD SYSTEM		AS NOTED

LB	DRAWN	DSM
DM	DSGN	CHKD
SMC	CHKD	APPD
DM	APPD	

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE IN WHOLE OR IN PART WITHOUT PERMISSION.
FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

KONNEX EMPIRE MINE

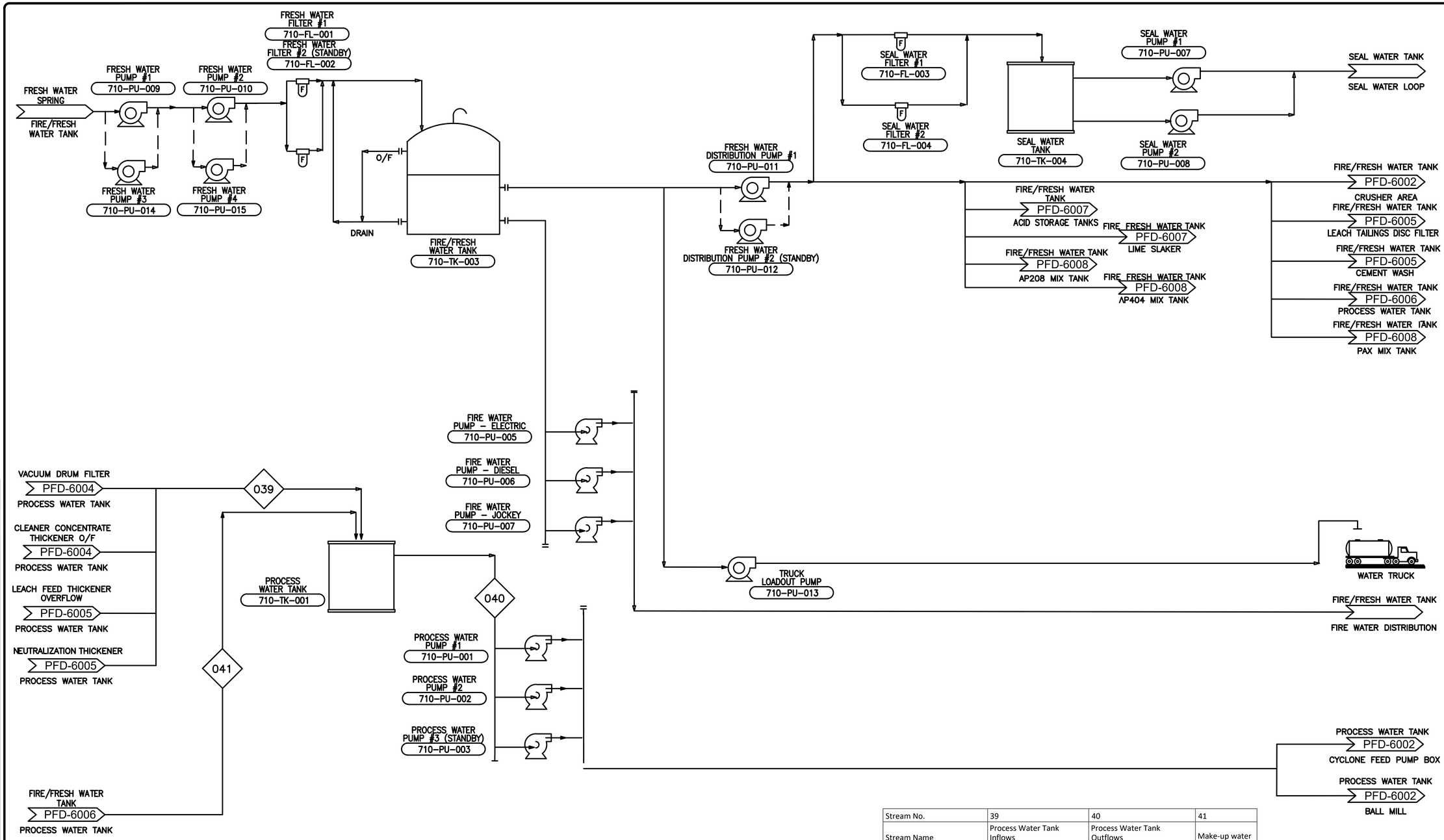
ACID LEACHING AND CEMENTATION

MACKAY, IDAHO

DRAWING NUMBER:

PFD-6005

SHEET 6 OF 10



Stream No.	39	40	41
Stream Name	Process Water Tank Inflows	Process Water Tank Outflows	Make-up water
Hours/day	24.0	24.0	24.0
Solids st/hr	0.0	0.0	0.0
% Solids	0.0	0.0	0.0
Water st/hr	289.4	331.6	42.2
Total Slurry Mass Flow st/hr	289.4	331.6	42.2
Slurry Volume Flow cuft/min	154.6	177.1	22.5
Slurry Volume Flow gpm	1156.5	1325.0	168.5
Copper Mass lb/hr	NA	NA	NA
Gold Mass troz/hr	NA	NA	NA
Silver Mass troz/hr	NA	NA	NA
Copper Grade			
Gold Grade troz/st			
Silver Grade troz/st			

ISSUE	DATE	DESCRIPTION	DRAWN	DSGN	CHKD	APPD
0	052424	ISSUED FOR USE	LB	DM	SMC	DM
COORD SYSTEM	AS NOTED					

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE IN WHOLE OR IN PART WITHOUT PERMISSION.
FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

KONNEX EMPIRE MINE

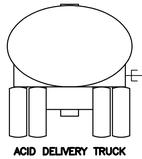
SEAL, FRESH, & FIRE WATER DISTRIBUTION

MACKAY, IDAHO

DRAWING NUMBER:

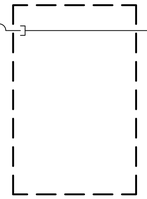
PFD-6006

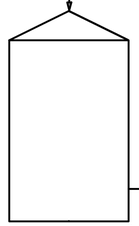
FIRE/FRESH WATER TANK
PFD-6006
ACID STORAGE TANKS



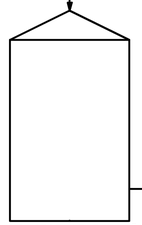
ACID DELIVERY TRUCK

TRUCK PUMP

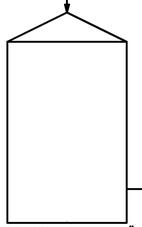




ACID STORAGE TANK #1
630-TK-002



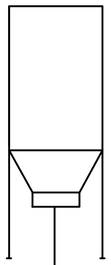
ACID STORAGE TANK #2
630-TK-003



ACID STORAGE TANK #3
630-TK-003

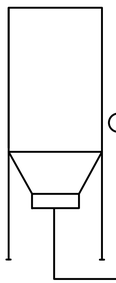
FRESH
ACID PUMP
630-PU-003

ACID MIX TANK
PFD-6005
DILUTE ACID STORAGE TANK



LIMESTONE SILO
520-TK-001

LIMESTONE SILO
PFD-6005
PUG MILL



CEMENT SILO
520-TK-002

CEMENT SILO
PFD-6005
PUG MILL

0	052424	ISSUED FOR USE		LB	DM	SMC	DM
ISSUE	DATE	DESCRIPTION		DRAWN	DSGN	CHKD	APPD

COORD SYSTEM: AS NOTED

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE, IN WHOLE OR IN PART WITHOUT PERMISSION.

FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

KONNEX EMPIRE MINE

ACID, LIMESTONE, AND CEMENT DISTRIBUTION

MACKAY, IDAHO

DRAWING NUMBER:

PFD-6007

SHEET 8 OF 10

FIRE/FRESH WATER TANK
PFD-6006
PAX MIX TANK

PAX TOTE

PAX MIX TANK AGITATOR
600-AG-001

PAX MIX TANK
600-TK-002

PAX MIX TANK PUMP
600-PU-004

PAX DAY TANK
600-TK-003

PAX DAY TANK PUMP
600-PU-005

PAX DAY TANK
PFD-6003
ROUGHER FLOTATION CONDITIONING TANK

FIRE/FRESH WATER TANK
PFD-6006
AP404 MIX TANK

AP404 TOTE

AP404 TOTE PUMP
600-PU-006

AP404 MIX TANK
600-TK-004

AP404 MIX TANK AGITATOR
600-AG-002

AP404 MIX TANK PUMP
600-PU-007

AP404 DAY TANK
600-TK-005

AP404 DAY TANK PUMP
600-PU-008

AP404 DAY TANK
PFD-6003
ROUGHER FLOTATION CONDITIONING TANK

FIRE/FRESH WATER TANK
PFD-6006
AP208 MIX TANK

AP208 TOTE

AP208 TOTE PUMP
600-PU-009

AP208 MIX TANK
600-TK-006

AP208 MIX TANK AGITATOR
600-AG-003

AP208 MIX TANK PUMP
600-PU-010

AP208 DAY TANK
600-TK-007

AP208 DAY TANK PUMP
600-PU-011

AP208 DAY TANK
PFD-6003
ROUGHER FLOTATION CONDITIONING TANK

AF-65 DRUM

AF-65 PUMP
600-PU-012

AF-65 DRUM
PFD-6003
ROUGHER FLOTATION CONDITIONING TANK

FATTY ACID DRUM

FATTY ACID PUMP
600-PU-013

FATTY ACID PUMP
PFD-6003
CARBONATE FLOTATION CONDITIONING TANK

ISSUE	DATE	DESCRIPTION	COORD SYSTEM
0	052424	ISSUED FOR USE	AS NOTED

LB	DRAWN	DM	DSGN
SMC	CHKD	DM	APPD

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE, IN WHOLE OR IN PART WITHOUT PERMISSION.

FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS

120 COMMERCE DRIVE
FORT COLLINS, CO 80528

KONNEX EMPIRE MINE

LIME SLAKER SYSTEM, AP404 SYSTEM, AP28 SYSTEM, AND MIBC DISTRIBUTION

MACKAY, IDAHO

DRAWING NUMBER:

PFD-6008

SHEET 9 OF 10

ATMOSPHERIC AIR

AIR COMPRESSOR SYSTEM
700–AC–001

AIR COMPRESSOR
PLANT AIR

DIESEL DELIVERY TRUCK

TRUCK PUMP

RED DYE DIESEL FUEL TANK
720–TK–001

RED DYE DIESEL PUMP
720–PU–001

VENDOR PACKAGE

RED DYE DIESEL FILTER
720–FL–001

RED DYE FUEL DISPENSER
720–DSP–001

DIESEL EXHAUST FLUID DELIVERY TRUCK

TRUCK PUMP

DIESEL EXHAUST FLUID TAN
720–TK–002

DIESEL EXHAUST FLUID PUMP
720–PU–002

DIESEL EXHAUST FLUID DISPENSER
720–DSP–002

ISSUE	DATE	DESCRIPTION	COORD SYSTEM
0	052424	ISSUED FOR USE	AS NOTED

DRAWN	DSGN	SMC	APPD
LB	DM	SMC	DM

THIS DRAWING IS THE PROPERTY OF FORTE DYNAMICS, INC. AND NO REPRODUCTION MAY BE MADE, IN WHOLE OR IN PART WITHOUT PERMISSION.

FIELD DEVIATIONS FROM THIS DESIGN ARE NOT PERMITTED WITHOUT DOCUMENTED APPROVAL BY THE ENGINEER.

FDI PROJECT NO: 134004

PREPARED BY:

FORTE DYNAMICS
120 COMMERCE DRIVE
FORT COLLINS, CO 80528

KONNEX EMPIRE MINE

COMPRESSED AIR AND DIESEL DISTRIBUTION SYSTEM

MACKAY, IDAHO

DRAWING NUMBER:

PFD-6009

SHEET 10 OF 10

APPENDIX B: MECHANICAL EQUIPMENT LIST

ASCENDING EQUIPMENT COUNT	EQUIPMENT NUMBER	PFD DRAWING	DESCRIPTION	TYPE	SIZE	DESIGN CAPACITY	POWER	CAPEX PAGE	QUOTES (Y/N)	BID DOCUMENT NUMBER	BIDS:	SELECTED VENDOR
1	110-RB-001	PFD-6002	Rock Breaker	Crushing	TBD	TBD	50 hp	Crushing	N			
2	110-FE-001	PFD-6002	Vibrating Grizzly Feeder	Crushing	36" W x 16ft L	220 dstph	25 hp	Crushing	Y	5000	FLS, Metso, Weir	Metso
3	110-CR-001	PFD-6002	Primary Crusher	Crushing	30 x 40 in	300/356 stph - rated / design	120 Hp	Crushing	Y	5000	FLS, Metso, Weir	FLS
4	110-MAG-001	PFD-6002	Belt Magnet	Crushing	Belt Width 36-48 in	TBD	7.5 hp	Crushing	Y			
5	110-CV-001	PFD-6002	Primary Crusher Discharge Conveyor	Crushing	20ft Length x 0ft Lift	220 dstph	30 hp	Crushing	TBD			
6	110-CV-002	PFD-6002	Secondary Crusher Feed Conveyor	Crushing	200ft Length x 40ft Lift	220 dstph	30 hp	Crushing	TBD			
7	120-CR-001	PFD-6002	Secondary Crusher	Crushing	4 ft	220 dstph	300 hp	Crushing	Y	5000	FLS, Metso, Weir	FLS
8	130-CV-001	PFD-6002	Crusher Screen Feed Conveyor	Crushing	240ft Length x 40ft Lift	332 dstph	30 hp	Crushing	TBD			
9	130-SR-001	PFD-6002	Crushing Plant Screen	Crushing	10ft W x 19ft L	440 dstph	20 hp	Crushing	Y	5000	FLS, Metso, Weir	FLS
10	130-CV-002	PFD-6002	Plant Screen Undersize Conveyor	Crushing	180ft Length x 500ft Lift	332 dstph	30 hp	Crushing	TBD			
11	130-CV-003	PFD-6002	Plant Screen Oversize Conveyor	Crushing	240ft Length x 55ft Lift	112 dstph	30 hp	Crushing	TBD			
12	140-CR-001	PFD-6002	Tertiary Crusher	Crushing	3 ft	112 dstph	200 hp	Crushing	Y	5000	FLS, Metso, Weir	FLS
13	140-CV-001	PFD-6002	Tertiary Crusher Discharge Conveyor	Crushing	245ft Length x 50ft Lift	112 dstph	30 hp	Crushing	TBD			
14	150-BN-001	PFD-6002	Fine Ore Bin With Feeder	Crushing	36 D x 57 H ft - 58,000 ft^3	2,000 Tons	15 hp	Crushing	Y			
15	150-FE-001	PFD-6002	Fine Ore Bin Discharge Belt Feeder	Crushing	TBD	220 dstph	TBD	Crushing	Y			
16	210-BN-001	PFD-6002	Grinding Media Storage Bin	Grinding	TBD	max. 3 inch grinding balls	NA	Grinding	Y			
17	210-FE-001	PFD-6002	Grinding Media Storage Bin Screw Feeder	Grinding	TBD	max. 3 inch grinding balls	2 hp	Grinding	Y			
18	210-CV-001	PFD-6002	Ball Mill Feed Conveyor	Grinding	100ft Length x 25ft Lift	167 dstph	30 hp	Grinding	TBD			
19	210-BM-001	PFD-6002	Ball Mill	Grinding	14.5ft D x 28ft L	Fresh Feed - 175 t/hr, Circulating Load - 399 t/hr	3,000 hp	Grinding	Y	5001	FLS, Metso	FLS
20	210-PU-001	PFD-6002	Cyclone Feed Pump #1	Grinding	TBD	Slurry 1117.2 st/hr (3296.8 gpm), % solids 52%	300 hp	Grinding	N			
21	210-PU-002	PFD-6002	Cyclone Feed Pump #2 (Standby)	Grinding	TBD	Slurry 1117.2 st/hr (3296.8 gpm), % solids 52%	300 hp	Grinding	N			
22	210-PB-001	PFD-6002	Cyclone Feed Pump Box	Grinding	11ft W x 13.5ft H x 11ft D	8,000 gallons	N/A	Grinding	N			
23	210-CY-001	PFD-6002	Cyclone Pack	Grinding	3 operating, 1 standby	Slurry 1117.2 st/hr (3296.8 gpm), % solids 52%	N/A	Grinding	Y	5003	FLS, Weir	FLS
24	310-TK-001	PFD-6003	Rougher Flotation Conditioning Tank	Flotation	9ft D x 9 ft H	3,300 gal	N/A	Flotation	Y	5006B	Yanke, GBI, SGS	Yanke
25	310-AG-001	PFD-6003	Rougher Float Conditioning Tank Agitator	Flotation	Shaft Length - 98", Impeller Diameter - 45"	TBD	5 hp	Flotation	Y	5005	Mixtec	Mixtec
26	310-FC-001	PFD-6003	Rougher Flotation Cell #1	Flotation	Total Bank Dimension - 44.85L x 14.31W x 24H ft	30m^3 (1,060 ft^3)	75 hp	Flotation	Y	5002	FLS, Metso	FLS
27	310-FC-002	PFD-6003	Rougher Flotation Cell #2	Flotation	Total Bank Dimension - 44.85L x 14.31W x 24H ft	30m^3 (1,060 ft^3)	75 hp	Flotation	Y	5002	FLS, Metso	FLS
28	310-FC-003	PFD-6003	Rougher Flotation Cell #3	Flotation	Total Bank Dimension - 44.85L x 14.31W x 24H ft	30m^3 (1,060 ft^3)	75 hp	Flotation	Y	5002	FLS, Metso	FLS
29	310-FC-004	PFD-6003	Rougher Flotation Cell #4	Flotation	Total Bank Dimension - 44.85L x 14.31W x 24H ft	30m^3 (1,060 ft^3)	75 hp	Flotation	Y	5002	FLS, Metso	FLS
30	300-BL-001	PFD-6003	Flotation Blower #1	Flotation		1500 ACFM @5.3 psig	150 hp	Flotation	N			
31	300-BL-002	PFD-6003	Flotation Blower #2 (Standby)	Flotation		1500 ACFM @5.3 psig	150 hp	Flotation	N			
32	310-PU-001	PFD-6003	Rougher Tailings Pump #1	Flotation	TBD	Slurry 1464.3 gpm, % solids 36%	40 hp	Flotation	N			
33	310-PU-002	PFD-6003	Rougher Tailings Pump #2 (Standby)	Flotation	TBD	Slurry 1464.3 gpm, % solids 36%	40 hp	Flotation	N			
34	310-PB-001	PFD-6003	Rougher Tailings Pump Box	Flotation	8ft W x 10.1ft H x 9ft D	3,800 gallons	N/A	Flotation	N			
35	310-PU-003	PFD-6003	Rougher Concentrate Pump	Flotation	TBD	Slurry 104.9 gpm, % solids 20%	10 hp	Flotation	N			
36	315-TK-001	PFD-6003	Carbonate Flotation Conditioning Tank	Flotation	9ft D x 9 ft H	3,300 gal	N/A	Flotation	Y	5006B	Yanke, GBI, SGS	Yanke
37	315-AG-001	PFD-6003	Carbonate Floatation Conditioning Tank Agitator	Flotation	Shaft Length - 98", Impeller Diameter - 45"	TBD	5 hp	Flotation	Y	5005	Mixtec	Mixtec
38	315-FC-001	PFD-6003	Carbonate Flotation Cell #1	Flotation	TBD	30m^3 (1,060 ft^3)	75 hp	Flotation	Y	5002	FLS, Metso	FLS
39	315-FC-002	PFD-6003	Carbonate Flotation Cell #2	Flotation	TBD	30m^3 (1,060 ft^3)	75 hp	Flotation	Y	5002	FLS, Metso	FLS
40	315-PB-001	PFD-6003	Carbonate Tailings Pump Box	Flotation	8ft W x 10.1ft H x 9ft D	3,800 gallons	N/A	Flotation	N			
41	315-PU-003	PFD-6003	Carbonate Concentrate Pump	Flotation	TBD	Slurry 104.9 gpm, % solids 20%	10 hp	Flotation	N			
42	315-PU-001	PFD-6003	Carbonate Tailings Pump #1	Flotation	TBD	Slurry 1464.3 gpm, % solids 36%	40 hp	Flotation	N			
43	315-PU-002	PFD-6003	Carbonate Tailings Pump #2 (Standby)	Flotation	TBD	Slurry 1464.3 gpm, % solids 36%	40 hp	Flotation	N			
44	320-FC-001	PFD-6004	First Stage Cleaner Flotation Cell #1	Flotation	Total Bank Dimension - 20L x 6.56W x 10.36H ft	2.8m^3 (100ft^3)	7.5 hp	Flotation	Y	5002	FLS, Metso	FLS
45	320-FC-002	PFD-6004	First Stage Cleaner Flotation Cell #2	Flotation	Total Bank Dimension - 20L x 6.56W x 10.36H ft	2.8m^3 (100ft^3)	7.5 hp	Flotation	Y	5002	FLS, Metso	FLS
46	320-FC-003	PFD-6004	First Stage Cleaner Flotation Cell #3	Flotation	Total Bank Dimension - 20L x 6.56W x 10.36H ft	2.8m^3 (100ft^3)	7.5 hp	Flotation	Y	5002	FLS, Metso	FLS
47	320-PU-001	PFD-6004	First Stage Cleaner Concentrate Pump	Flotation	TBD	Slurry 28.4 gpm, % solids 15%	10 hp	Flotation	N			
48	320-PB-001	PFD-6004	Cleaner Tailings Pump Box	Flotation	4ft W x 5.5ft H x 4ft D	465 gallons	N/A	Flotation	N			
49	320-PU-003	PFD-6004	Cleaner Tailings Pump #1	Flotation	TBD	Slurry 76.4 gpm, % solids 22%	10 hp	Flotation	N			
50	320-PU-004	PFD-6004	Cleaner Tailings Pump #2 (Standby)	Flotation	TBD	Slurry 76.4 gpm, % solids 22%	10 hp	Flotation	N			
51	325-FC-001	PFD-6004	Second Stage Cleaner Flotation Cell #1	Flotation	TBD	2.8m^3 (100ft^3)	7.5 hp	Flotation	Y	5002	FLS, Metso	FLS
52	325-PU-001	PFD-6004	Second Stage Cleaner Concentrate Pump	Flotation	TBD	TBD	10 hp	Flotation	N			
53	330-THK-001	PFD-6004	Cleaner Concentrate Thickener	Flotation	9.8ft D x 5.2ft H	Slurry 28.4 gpm, % solids 15%	1 hp	Flotation	Y	5004	FLS, Takraf, Westech	FLS
54	330-PU-001	PFD-6004	Cleaner Concentrate Thickener U/F Pump #1	Flotation	TBD	Slurry 4.79 gpm, % solids 65%	10 hp	Flotation	N			
55	330-PU-002	PFD-6004	Cleaner Concentrate Thickener U/F Pump #2 (Standby)	Flotation	TBD	Slurry 4.79 gpm, % solids 65%	10 hp	Flotation	N			
56	340-FL-001	PFD-6004	Concentrate Vacuum Drum Filter Skid	Flotation	38.9ft L x 15ft W	144.6 ft^2 filter area	2 hp	Flotation	Y	5007	FLS, Metso, Westech	Metso
57	340-VS-001	PFD-6004	Vacuum Station	Flotation	TBD	TBD	N/A	Flotation	Y	5007	FLS, Metso, Westech	Metso
58	340-TK-001	PFD-6004	Filtrate Reciever	Flotation	2'D x 5'H	TBD	N/A	Flotation	Y	5007	FLS, Metso, Westech	Metso
59	340-BN-001	PFD-6004	Roll Away Bins	Flotation	TBD	926 lb/h	N/A	Flotation	N			
60	340-CV-001	PFD-6004	Vacuum Drum Filter Cake Conveyor	Flotation	25ft Length x 15ft Lift	1.2 dstph	5 hp	Flotation	TBD	5007	FLS, Metso, Westech	Metso
61	410-THK-001	PFD-6005	Leach Circuit Feed Thickener	Leach	72.2ft D x 6.2ft H	161 st/hr	15 hp	Leach	Y	5004	FLS, Takraf, Westech	FLS
62	410-PU-001	PFD-6005	Thickener Underflow Pump #1	Leach	TBD	Slurry 847 gpm, % solids 55%	40 hp	Leach	N			
63	410-PU-002	PFD-6005	Thickener Underflow Pump #2 (Standby)	Leach	TBD	Slurry 847 gpm, % solids 55%	40 hp	Leach	N			
64	420-TK-001	PFD-6005	Copper Leach Tank #1	Leach	29ft D x 29 ft H	17,750 ft3	N/A	Leach	Y	5010	Yanke, GBI	GBI
65	420-TK-002	PFD-6005	Copper Leach Tank #2	Leach	29ft D x 29 ft H	17,750 ft3	N/A	Leach	Y	5010	Yanke, GBI	GBI
66	420-TK-003	PFD-6005	Copper Leach Tank #3	Leach	29ft D x 29 ft H	17,750 ft3	N/A	Leach	Y	5010	Yanke, GBI	GBI
67	420-AG-001	PFD-6005	Copper Leach Tank Agitator #1	Leach	Shaft Length - 234", Impeller Diameter - 144"	TBD	15 hp	Leach	Y	5005	Mixtec	Mixtec
68	420-AG-002	PFD-6005	Copper Leach Tank Agitator #2	Leach	Shaft Length - 234", Impeller Diameter - 144"	TBD	15 hp	Leach	Y	5005	Mixtec	Mixtec
69	420-AG-003	PFD-6005	Copper Leach Tank Agitator #3	Leach	Shaft Length - 234", Impeller Diameter - 144"	TBD	15 hp	Leach	Y	5005	Mixtec	Mixtec
70	420-PU-001	PFD-6005	Leach Tank Transfer Pump #1	Leach	TBD	Slurry 1106.3 gpm, % solids 45%	40 hp	Leach	N			
71	420-PU-002	PFD-6005	Leach Tank Transfer Pump #2 (Standby)	Leach	TBD	Slurry 1106.3 gpm, % solids 45%	40 hp	Leach	N			
72	520-FL-001	PFD-6005	Leach Tailings Disc Filter #1	Cementation & Tails	8 Discs @ 8ft - 10in	Filtration Area - 900 ft2	TBD	Cementation & Tails	Y			
73	520-FL-002	PFD-6005	Leach Tailings Disc Filter #2	Cementation & Tails	8 Discs @ 8ft - 10in	Filtration Area - 900 ft2	TBD	Cementation & Tails	Y			
74	520-TK-001	PFD-6005	Re-Pulp Tank	Cementation & Tails	9ft D x 9 ft H	3,300 gal	N/A	Cementation & Tails	Y	5006B	Yanke, GBI, SGS	Yanke
75	520-AG-001	PFD-6005	Re-Pulp Tank Agitator	Cementation & Tails	Shaft Length - 98", Impeller Diameter - 45"	TBD	5 hp	Cementation & Tails	Y	5005	Mixtec	Mixtec
76	520-VS-001	PFD-6005	Vaccum Station #1	Cementation & Tails	*By Vendor - Part of the Disc Filter	TBD	200 hp	Cementation & Tails	Y			
77	520-VS-002	PFD-6005	Vaccum Station #2	Cementation & Tails	*By Vendor - Part of the Disc Filter	TBD	200 hp	Cementation & Tails	Y			
78	520-PU-001	PFD-6005	Re-Pulp Tank Pump #1	Cementation & Tails	TBD	TBD	15 hp	Cementation & Tails	Y			
79	520-PU-002	PFD-6005	Re-Pulp Tank Pump #2 (Standby)	Cementation & Tails	TBD	TBD	15 hp	Cementation & Tails	Y			
80	520-PU-003	PFD-6005	Mother Liquor Pump #1	Cementation & Tails	TBD	Slurry 196.6 st/h (785.5 gpm), % solids 0%	30 hp	Cementation & Tails	N	5008	FLS, Takraf	Takraf

81	520-PU-004	PFD-6005	Mother Liquor Pump #2 (Standby)	Cementation & Tails	TBD	Slurry 196.6 st/h (785.5 gpm), % solids 0%	30 hp	Cementation & Tails	N	5008	FLS, Takraf	Takraf
82	520-CV-001	PFD-6005	Filter Cake Conveyor	Cementation & Tails	10ft Length x 0ft Lift	Slurry Mass Flowrate 205 st/hr, % solids 85%	5 hp	Cementation & Tails	TBD			
83	520-RS-001	PFD-6005	Dry Stack Taillings Radial Stacker	Cementation & Tails	30"x80' Portable Conveyor Stacker	Slurry Mass Flowrate 205 st/hr, % solids 85%	25 hp	Cementation & Tails	N			
84	520-CV-003	PFD-6005	Dry Stack Tailings Grasshopper Conveyors	Cementation & Tails	30" Portable Jump Conveyor - 100ft length	Slurry Mass Flowrate 205 st/hr, % solids 85%	25 hp	Cementation & Tails	N			
85	520-PUG-001	PFD-6005	PUG Mill	Cementation & Tails	60" W X 11ft L	205 stph	120 hp	Cementation & Tails	Y	5014	Feeco, Scott	Scott Equipment
86	520-TK-001	PFD-6007	Limestone Silo	Cementation & Tails	12ft D x 55.5 ft H	180 - 200 st	30.5 hp	Cementation & Tails	Y			
87	520-TK-002	PFD-6007	Cement Silo	Cementation & Tails	12ft D x 55.5 ft H	180 - 200 st	30.5 hp	Cementation & Tails	Y			
88	530-TK-001	PFD-6005	Cementation Surge Tank	Cementation & Tails	14ft D x 14ft H	13,800 gal	N/A	Cementation & Tails	Y	5006A	Yanke, GBI	Yanke
89	530-PU-001	PFD-6005	Cementation Vessel Feed Pump	Cementation & Tails	TBD	Slurry 862.9 gpm	30 hp	Cementation & Tails	N			
90	530-PU-002	PFD-6005	Cementation Reactor Underflow Pump	Cementation & Tails	TBD	Slurry 221 gpm	10 hp	Cementation & Tails	N			
91	530-TK-002	PFD-6005	Cementation Reactor Tank	Cementation & Tails	14ft D x 14ft H	13,800 gal	N/A	Cementation & Tails	Y	5006A	Yanke, GBI	Yanke
92	530-TK-003	PFD-6005	Cementation Tank #1	Cementation & Tails	10ft D x 17ft H	1,730 ft3	N/A	Cementation & Tails	Y	5009	CGS	CGS
93	530-TK-004	PFD-6005	Cementation Tank #2 (Standby)	Cementation & Tails	10ft D x 17ft H	1,730 ft3	N/A	Cementation & Tails	Y	5009	CGS	CGS
94	530-PU-003	PFD-6005	Cementation Tank Underflow Pump	Cementation & Tails	TBD	Slurry 221 gpm	10 hp	Cementation & Tails	N			
95	530-SR-001	PFD-6005	Cement Copper Wash Screen	Cementation & Tails	4ft x 8ft - horizontal vibrating	Slurry Mass Flow rate 0.61 st/hr, 85% solids	10 hp	Cementation & Tails	TBD			
96	530-HO-001	PFD-6005	Cementation Overhead Crane	Cementation & Tails	TBD	0.5 Ton	100 hp	Cementation & Tails	N			
97	530-MAG-001	PFD-6005	Cementation Overhead Magnet	Cementation & Tails	10,500 lb capacity	10,500 lb capacity	7.5 hp	Cementation & Tails	N			
98	530-PU-004	PFD-6005	Wash Screen Underflow Pump #1	Cementation & Tails	TBD	221 gpm	1 hp	Cementation & Tails	N			
99	530-PU-005	PFD-6005	Wash Screen Underflow Pump #2 (Standby)	Cementation & Tails	TBD	221 gpm	1 hp	Cementation & Tails	N			
100	500-PU-001		Talings Pond Pump #1	Cementation & Tails	TBD	111 gpm max	15 hp	Cementation & Tails	N			
101	500-PU-002		Talings Pond Pump #2 (Standby)	Cementation & Tails	TBD	111 gpm max	15 hp	Cementation & Tails	N			
102	710-PU-001	PFD-6006	Process Water Pump #1	Site Utilities	TBD	700 gpm @ 50'	25 hp	Site Utilities	N			
103	710-PU-002	PFD-6006	Process Water Pump #2	Site Utilities	TBD	700 gpm @ 50'	25 hp	Site Utilities	N			
104	710-PU-003	PFD-6006	Process Water Pump #3 (Standby)	Site Utilities	TBD	700 gpm @ 50'	25 hp	Site Utilities	N			
105	710-PU-004	PFD-6006	Crusher Fire Water Pump Electric	Site Utilities	TBD	1500 gpm @ 300'	200 hp	Site Utilities	TBD			
106	710-PU-005	PFD-6006	Crusher Fire Water Pump Diesel	Site Utilities	TBD	1500 gpm @ 300'	NA	Site Utilities	TBD			
107	710-PU-006	PFD-6006	Crusher Fire Water Pump Jockey	Site Utilities	TBD	10 gpm @ 300'	2 hp	Site Utilities	TBD			
108	710-PU-007	PFD-6006	Seal Water Pump #1	Site Utilities	TBD	Estimated 70 gpm	100 hp	Site Utilities	N			
109	710-PU-008	PFD-6006	Seal Water Pump #2	Site Utilities	TBD	Estimated 70 gpm	100 hp	Site Utilities	N			
110	710-PU-009	PFD-6006	Freshwater Pump #1	Site Utilities	TBD	200 gpm @ 2500'	250 hp	Site Utilities	Y	5013	Pumps Plus, CHSpencer	
111	710-PU-010	PFD-6006	Freshwater Pump #2	Site Utilities	TBD	200 gpm @ 2500'	250 hp	Site Utilities	Y	5013	Pumps Plus, CHSpencer	
112	710-PU-014	PFD-6006	Freshwater Pump #3 (Standby)	Site Utilities	TBD	200 gpm @ 2500'	250 hp	Site Utilities	Y	5013	Pumps Plus, CHSpencer	
113	710-PU-015	PFD-6006	Freshwater Pump #4 (Standby)	Site Utilities	TBD	200 gpm @ 2500'	250 hp	Site Utilities	Y	5013	Pumps Plus, CHSpencer	
114	710-PU-011	PFD-6006	Fresh Water Distribution Pump #1	Site Utilities	TBD	850 gpm @ 100' TDH	40 hp	Site Utilities	N			
115	710-PU-012	PFD-6006	Fresh Water Distribution Pump #2 (Standby)	Site Utilities	TBD	850 gpm @ 100' TDH	40 hp	Site Utilities	N			
116	710-PU-013	PFD-6006	Truck Loadout Pump	Site Utilities	TBD	1000 gpm @ 50'	25 hp	Site Utilities	N			
117	710-TK-001	PFD-6006	Process Water Tank	Site Utilities	16ft D x 18ft H	24,000 gal	N/A	Site Utilities	Y	5006A	Yanke, GBI	Yanke
118	710-TK-003	PFD-6006	Fire/Fresh Water Tank	Site Utilities	40ft D x 40ft H	357,000 gal	N/A	Site Utilities	Y	5006A	Yanke, GBI	Yanke
119	710-TK-004	PFD-6006	Seal Water Tank	Site Utilities	9ft D x 9ft H	3,300 gal	N/A	Site Utilities	Y	5006B	Yanke, GBI, SGS	Yanke
120	710-FL-001	PFD-6006	Fresh Water Filter #1	Site Utilities	TBD	200 gpm	N/A	Site Utilities	N			
121	710-FL-002	PFD-6006	Fresh Water Filter #2	Site Utilities	TBD	200 gpm	N/A	Site Utilities	N			
122	710-FL-003	PFD-6006	Seal Water Filter #1	Site Utilities	TBD	70 gpm	N/A	Site Utilities	N			
123	710-FL-004	PFD-6006	Seal Water Filter #2	Site Utilities	TBD	70 gpm	N/A	Site Utilities	N			
124	610-SK-001	PFD-6005	Water Treatment Skid	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
125	610-THK-001	PFD-6005	Neutralization Clarifier	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
126	610-TK-001	PFD-6005	Iron Oxididation Tank	Mill Reagents	TBD	TBD	N/A	Mill Reagents	TBD			
127	610-TK-002	PFD-6005	Neutralization Reactor Tank	Mill Reagents	TBD	TBD	N/A	Mill Reagents	TBD			
128	610-TK-003	PFD-6005	Sludge Tank	Mill Reagents	TBD	TBD	N/A	Mill Reagents	TBD			
129	610-AG-001	PFD-6005	Iron Oxididation Tank Agitator	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
130	610-AG-002	PFD-6005	Neutralization Reactor Tank Agitator	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
131	610-AG-003	PFD-6005	Sludge Tank Agitator	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
132	610-PU-001	PFD-6005	Neautralization Clarifier Overrflow Pump	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
133	610-PU-002	PFD-6005	Neutralization Clarifier Underflow Pump #1	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
134	610-PU-003	PFD-6005	Neutralization Clarifier Underflow Pump #2 (Standby)	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
135	610-PU-004	PFD-6005	Sludge Tank Pump #1	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
136	610-PU-005	PFD-6005	Sludge Tank Pump #2 (Standby)	Mill Reagents	TBD	TBD	TBD	Mill Reagents	TBD			
137	630-TK-002	PFD-6007	Acid Storage Tank #1	Mill Reagents	18ft D x 18ft H	30,500 gal	N/A	Mill Reagents	Y	5006A	Yanke, GBI	Yanke
138	630-TK-003	PFD-6007	Acid Storage Tank #2	Mill Reagents	18ft D x 18ft H	30,500 gal	N/A	Mill Reagents	Y	5006A	Yanke, GBI	Yanke
139	630-TK-004	PFD-6007	Acid Storage Tank #3	Mill Reagents	18ft D x 18ft H	30,500 gal	N/A	Mill Reagents	Y	5006A	Yanke, GBI	Yanke
140	630-TK-001	PFD-6005	Dilute Acid Storage Tank	Mill Reagents	10ft D x 10ft H	4,700 gal	N/A	Mill Reagents	Y	5006A	Yanke, GBI	Yanke
141	630-PU-001	PFD-6005	Dilute Acid Pump #1	Mill Reagents	TBD	250 gpm	10 hp	Mill Reagents	N			
142	630-PU-002	PFD-6005	Dilute Acid Pump #2 (Standby)	Mill Reagents	TBD	250 gpm	10 hp	Mill Reagents	N			
143	630-PU-003	PFD-6007	Fresh Acid Pump	Mill Reagents	TBD	25 gpm	1 hp	Mill Reagents	N			
144	600-TK-002	PFD-6008	Pax Mix Tank	Mill Reagents	5ft D x 5ft H	580 gal	N/A	Mill Reagents	Y	5006B	Yanke, GBI, SGS	Yanke
145	600-TK-003	PFD-6008	Pax Day Tank	Mill Reagents	5ft D x 5ft H	580 gal	N/A	Mill Reagents	Y	5006B	Yanke, GBI, SGS	Yanke
146	600-TK-004	PFD-6008	AP404 Mix Tank	Mill Reagents	5ft D x 5ft H	580 gal	N/A	Mill Reagents	Y	5006B	Yanke, GBI, SGS	Yanke
147	600-TK-005	PFD-6008	AP404 Day Tank	Mill Reagents	5ft D x 5ft H	580 gal	N/A	Mill Reagents	Y	5006B	Yanke, GBI, SGS	Yanke
148	600-TK-006	PFD-6008	AP208 Mix Tank	Mill Reagents	5ft D x 5ft H	580 gal	N/A	Mill Reagents	Y	5006B	Yanke, GBI, SGS	Yanke
149	600-TK-007	PFD-6008	AP208 Day Tank	Mill Reagents	5ft D x 5ft H	580 gal	N/A	Mill Reagents	Y	5006B	Yanke, GBI, SGS	Yanke
150	600-AG-001	PFD-6008	Pax Mix Tank Agitator	Mill Reagents	Tank Size - 5ft D x 5ft H	TBD	TBD	Mill Reagents	N			
151	600-AG-002	PFD-6008	AP404 Mix Tank Agitator	Mill Reagents	Tank Size - 5ft D x 5ft H	TBD	TBD	Mill Reagents	N			
152	600-AG-003	PFD-6008	AP208 Mix Tank Agitator	Mill Reagents	Tank Size - 5ft D x 5ft H	TBD	TBD	Mill Reagents	N			
153	600-PU-004	PFD-6008	PAX Mix Tank Pump	Mill Reagents	TBD	15 gpm	1 hp	Mill Reagents	N			
154	600-PU-005	PFD-6008	PAX Day Tank Pump	Mill Reagents	TBD	TBD	0.05 hp	Mill Reagents	N			
155	600-PU-006	PFD-6008	AP404 Tote Pump	Mill Reagents	TBD	15 gpm	1 hp	Mill Reagents	N			
156	600-PU-007	PFD-6008	AP404 Mix Tank Pump	Mill Reagents	TBD	15 gpm	1 hp	Mill Reagents	N			
157	600-PU-008	PFD-6008	AP404 Day Tank Pump	Mill Reagents	TBD	TBD	0.05 hp	Mill Reagents	N			
158	600-PU-009	PFD-6008	AP208 Tote Pump	Mill Reagents	TBD	15 gpm	1 hp	Mill Reagents	N			
159	600-PU-010	PFD-6008	AP208 Mix Tank Pump	Mill Reagents	TBD	15 gpm	1 hp	Mill Reagents	N			
160	600-PU-011	PFD-6008	AP208 Day Tank Pump	Mill Reagents	TBD	TBD	0.05 hp	Mill Reagents	N			
161	600-PU-012	PFD-6008	AF-65 Pump	Mill Reagents	TBD	TBD	0.05 hp	Mill Reagents	N			
162	600-PU-013	PFD-6008	Fatty Acid Pump	Mill Reagents	TBD	TBD	0.05 hp	Mill Reagents	N			
163	700-AC-001	PFD-6009	Air Compressor System	Site Utilities	TBD	250 CFM	75 hp	Site Utilities	N			
164	720-TK-001	PFD-6009	Red Dye Diesel Fuel Tank	Site Utilities	40ft L x 8ft W x 9.5ft H	15,000 gal	N/A	Site Utilities	N			
165	720-PU-001	PFD-6009	Red Dye Diesel Pump	Site Utilities	TBD	30 gpm	TBD	Site Utilities	N			
166	720-FL-001	PFD-6009	Red Dye Diesel Filter	Site Utilities	10 micron	25 gpm	N/A	Site Utilities	N			
167	720-DSP-001	PFD-6009	Red Dye Fuel Dispenser	Site Utilities	TBD	25 gpm	TBD	Site Utilities	N			
168	720-DSP-002	PFD-6009	Diesel Exhaust Fluid Dispenser	Site Utilities	TBD	10 gpm	TBD	Site Utilities	N			
169	720-TK-002	PFD-6009	Diesel Exhaust Fluid Tank	Site Utilities	44" W x 150" L x 106" H	1,000 gal	N/A	Site Utilities	N			
170	720-PU-002	PFD-6009	Diesel Exhaust Fluid Pump	Site Utilities	TBD	10 gpm	TBD	Site Utilities	N			
171	720-SC-001		Truck Scale	Site Utilities	80'L x 11'W	200,000 lbs	TBD	Site Utilities	N			
172	700-TK-001		Prill Silo	Site Utilities	12ft D x 55.5 ft H	180 - 200 st	30.5 hp	Site Utilities	N			

APPENDIX C: EMPIRE PHASE I MET REPORT

Konnex Resources – Empire Mine
Metallurgical Study Phase I Results
Project No. 23009

Prepared by:

FORTE ANALYTICAL, LLC
120 Commerce Drive
Units 3 & 4
Fort Collins, CO 80524

Table of Contents

1. Introduction 5

2. Historical Studies 5

3. Composite Preparation and Characterization 5

4. Comminution Tests 7

5. Gravity Testing 8

6. Ammonium Thiosulfate Leaching 11

7. Sulfuric Acid Leaching 14

8. Flotation Testing 16

8.1 Rougher Flotation Tests 16
8.2 Cleaner Flotation Tests 18
8.3 Locked-Cycle Tests 19
8.4 Open-Cycle Rougher and Cleaner Flotation Tests for Variable Feed Materials 20

9. Smelter Penalty Analyses of Flotation Concentrates 23

10. Thickening Test 24

11. Filtration Tests 25

12. Cementation Testing 25

13. Evaluation of Alternative Processing Flowsheets 26

13.1 Gravity/Flotation Process 26
13.2 Agitated ATS and Acid Leach Process 27
13.3 Flotation/Acid Leach of Tailing 29

14. Conclusions 31

15. Recommendations 31

Figures

Figure 1: Flowsheet for Gravity/Flotation Testing of Master Composites (Tests 1-8) 9
Figure 2: Gravity Testing Flowsheet for High Grade Magnetite Master Composites (Tests 9-11) 10
Figure 3: Rougher and Cleaner Open-Circuit Test Flowsheet 18
Figure 4: Locked-Cycle Test Flowsheet 19
Figure 5: Copper Recovery as Function of Feed Grade 22
Figure 6: Gold Recovery as Function of Feed Grade 23
Figure 7: Simplified Gravity/Flotation Flowsheet 27
Figure 8: Simplified ATS/Acid Leach Flowsheets 28
Figure 9: Simplified Flotation/Acid Leach Process Flowsheet 30

Tables

Table 1: Head Analyses of Composite Samples 6
Table 2: Whole Rock Analyses of Endo/Exo and Magnetite Master Composites 7
Table 3: Comminution Test Results 8
Table 4: Indirect Bond's Ball Mill Work Indices 8
Table 5: Gravity Test Results for Endo/Exo and Magnetite Master Composites 9
Table 6: Gravity Test Results for High Grade Magnetite Master Composite 10
Table 7: Ammonium Thio-Sulfate Leach (ATS) at Varying Grind Sizes for Endo/Exo and Magnetite Master Composites 12
Table 8: Effect of ATS Concentration on Metal Extraction at P_{80} of 65 mesh for Endo/Exo and Magnetite Master Composites 12
Table 9: Effect of Recycling ATS on Metal Extraction at P_{80} of 100 Mesh for Endo/Exo and Magnetite Composite 13
Table 10: ATS Testing of Variability in Flotation Tailing Samples 13
Table 11: Summary of Acid Leach Test on Flotation Tailings at Varying Grind Sizes 15
Table 12: Summary of Acid Leach Test on Flotation Tailings for Variability Samples 15
Table 13: Summary of Flotation Test Results for Endo/Exo and Magnetite Master Composites for Varying Grind Sizes 17
Table 14: Summary of Flotation Test Results for Endo/Exo and Magnetite Master Composites for Varying Grind Sizes and Sulfidization 17
Table 15: Rougher Flotation Test at P_{80} of 100 Mesh in One Cubic Foot Flotation Cell for Endo/Exo Master Composite (Test No. 17) 18
Table 16: Cleaner Flotation Tests With/Without Regrind of Rougher Concentrate from Flotation Test No. 17 19
Table 17: Summary of Locked-Cycle Test Results 20
Table 18: Flotation Test Results for Variability Samples 21
Table 19: Cleaner Flotation Concentrate Grades for Variability Samples 21
Table 20: Smelter Penalty Analyses of Copper Concentrates 23
Table 21: Chemical Penalty Analyses of Copper Concentrates 24
Table 22: Thickening Test Results 24
Table 23: Summary of Vacuum and Pressure Filtration Results 25
Table 24: Solution Analyses of Pre- and Post-Cementation for Endo/Exo Leach Test 26
Table 25: Analyses of Cemented Copper 26
Table 26: Gravity and Flotation Results for the Two Master Composites at P_{80} = 100 mesh 27
Table 27: Combined Results for Two-Stage Leach Process at P_{80} = 100 mesh 28
Table 28: Metal Recoveries for Flotation/Acid Leach Process for Endo/Exo Master Composite 31

Appendices

Appendix A: Preparation of Composite Samples
Appendix B: Head Analyses of Composite Samples
Appendix C: Comminution Test Results
Appendix D: Gravity Test Results
Appendix E: ATS Test Results
Appendix F: Acid Leach Test Results
Appendix G: Flotation Test Results
Appendix H: Locked-Cycle Test Results
Appendix I: Copper Concentrate Analyses
Appendix J: Thickening Test Results
Appendix K: Filtration Test Results
Appendix L: Distribution of Gold in Flotation Tailing
Appendix M: Copper Cementation Test Results

1. INTRODUCTION

Konnex Resources Inc. (Konnex) contracted Forte Analytical / Forte Dynamics (Forte) to undertake metallurgical test work with the primary objective of developing a techno-economically viable process flowsheet for recovering copper and precious metals from Phoenix Copper's Empire Mine without using sodium cyanide.

This report discusses the test procedures and results obtained in the study.

2. HISTORICAL STUDIES

The metallurgical test work has been performed on samples from the Empire Mine by several companies since 1997. Several processing options have been proposed by the investigators. These options include the following:

- Two-stage agitated leach process (acid leach/cyanide leach) for the recovery of copper and precious metals.
- Two-stage acid heap leach followed by cyanide agitated leach process for recovering copper and precious metals.
- One-stage agitated leach process using ammonium thiosulfate (ATS) to recover copper, gold, silver, and zinc.

3. COMPOSITE PREPARATION AND CHARACTERIZATION

Forte received 1/2 core from nine holes drilled at site in 2022. The location and description of these holes and intervals are given in Appendix A.

There are three geological domains – Endoskarn, Exoskarn, and Magnetite. Since Endoskarn and Exoskarn have similar metallurgical responses, a decision was made to prepare two master composites, Endo/Exo and Magnetite. A high-grade Magnetite composite, assaying over 2 g/t Au (0.0584 opt) was also received for the study. In addition, +2 inch rocks were received for crusher work index determination.

Each interval was crushed to nominal 3/4 inch and split into two fractions. One fraction was saved and the other fraction was used for preparing a composite sample. This procedure was utilized for both Endo/Exo and Magnetite composites.

The composite samples were blended and a split taken out for abrasion index determination. The remaining composite samples were crushed to P_{100} of 6 mesh, blended, and split into 1-kg and 10-kg charges for metallurgical studies.

One 10-kg sample of each composite was taken for Bond's ball mill work index (BWi) determination. One 1-kg charge was pulverized and splits were submitted for head analyses and whole rock analyses.

The analytical results for the master composites are given in Appendix B and summarized in Table 1 and Table 2.

.

Table 1: Head Analyses of Composite Samples

Element	Endo/Exo	Magnetite
Au, ppm		
Assay #1	0.16	0.14
Assay #2	0.12	0.15
Average	0.14	0.145
Ag, ppm		
Assay #1	22.7	13.7
Assay #2	14.7	15.0
Average	18.7	14.3
Cu_{Total}	0.43	0.75

Table 2: Whole Rock Analyses of Endo/Exo and Magnetite Master Composites

Element	Endo/Exo	Magnetite
SiO_2	53.31	43.95
Al_2O_3	10.04	5.92
Fe_2O_3	12.13	34.27
CaO	14.29	6.50
MgO	1.62	0.59
Na_2O	0.91	0.07
K_2O	2.62	0.59
TiO_2	0.37	0.29
MnO	0.38	0.54
SrO	0.02	<0.01
BaO	0.07	0.03
Cr_3O_3	0.01	0.02
P_2O_5	0.30	0.55
ppm		
Ag	19.4	19.4
As	112	316
Ba	606	286
Bi	<10	90
Cd	23	51
Co	23	41
Cr	96	98
Cu	4300	7400
Mo	62	177
Ni	25	45
Pb	138	177
Sr	261	87
V	62	77
W	174	193
Zn	3200	4600

These results indicate the following:

- The Endo/Exo master composite assayed 0.43% Cu, 0.14 ppm Au and 18.7 ppm Ag.
- The Magnetite master composite assayed 0.75% Cu, 0.145 ppm Au and 14.3 ppm Ag.

4. COMMINUTION TESTS

The comminution tests were performed on the two master composites. These tests included crusher work index, abrasion index, and Bond's ball mill work index. In addition, indirect ball mill work indices were evaluated for seven variability samples.

The comminution test results are given in Appendix C and summarized in Table 3 and Table 4.

Table 3: Comminution Test Results

Parameter	Endo/Exo Composite	Magnetite Composite
Crusher Work Index (CWi, kwh/st)	14.23	15.63
Abrasion Index (Ai)	0.0773	0.1566
Bond Ball Mill Work Index (BWi, kwh/st)	15.29	14.08

Table 4: Indirect Bond's Ball Mill Work Indices

Sample	Ore Type	BWi (kwh/st)
Interval 2200102 Hole KXD 22-03	Exo	14.33
Interval 2200101/103 Hole KXD 22-03	Exo	15.16
Interval 2200298/667 Holes KXD 22-06/09	Endo	13.71
Interval 2200104/596 Holes KXD 22-03/08	Endo	14.55
Interval 2200291 Hole KXD 22-06	Endo	11.84
Interval 2200679 Holes KXD 22-01/09	Endo	10.66
Interval 2200127/128 Hole KXD 22-03	Endo	18.39

These results indicate the following:

- Crusher work index (CWi) was 14.23 kwh/st for Endo/Exo composite and 15.63 kwh/st for Magnetite composite.
- Abrasion index (Ai) was 0.0773 for Endo/Exo composite and 0.1566 for Magnetite composite. Ai of more than 0.3 is considered to be abrasive.
- Bond's ball mill work index (BWi) was 15.29 kwh/st for Endo/Exo composite and 14.08 kwh/st for Magnetite composite.

The indirect BWi for the variability samples ranged from 11.84 to 18.39 kwh/st. The majority of the samples had a BWi of less than 15.29 kwh/st.

5. GRAVITY TESTING

A series of gravity tests were performed with the primary objective of recovering gold. The gravity tails were floated. These tests were performed with the Endo/Exo and Magnetite Master Composites (MC) at varying grind sizes.

The test flowsheet is shown in Figure 1. The test data are given in Appendix D and the results are summarized in Table 5.

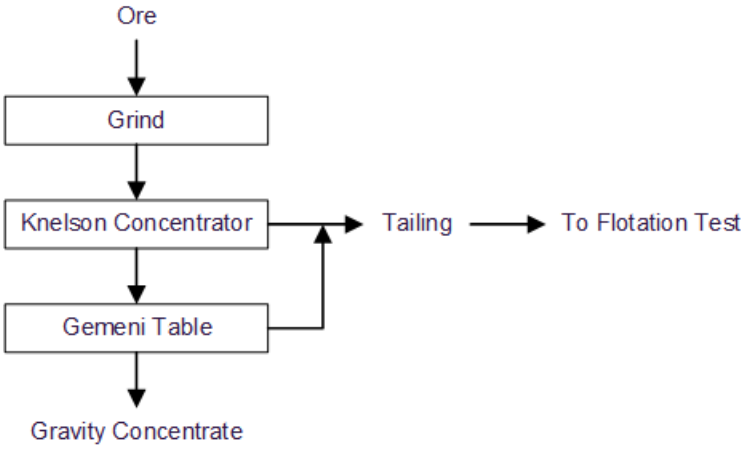


Figure 1: Flowsheet for Gravity/Flotation Testing of Master Composites (Tests 1-8)

Table 5: Gravity Test Results for Endo/Exo and Magnetite Master Composites

Test No.	Composite	Grind Size P_{80} Mesh	Cleaner Gravity Recovery %				Gravity Concentrate Grade			Calculated Feed Grade		
			Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
1	Endo/Exo MC	65	1.3	16.7	9.1	5.9	2.05	131	2.11	0.16	19	0.47
2	Endo/Exo MC	100	0.6	10.8	3.9	3.8	2.75	156	2.83	0.16	24.5	0.46
3	Endo/Exo MC	150	0.2	15.0	4.1	1.9	14.4	534	5.58	0.15	20.7	0.47
4	Endo/Exo MC	200	0.1	15.1	2.1	1.9	29.3	479	10.0	0.17	19.5	0.44
5	Magnetite MC	65	1.3	5.8	1.3	0.4	0.71	17.3	0.21	0.16	16.9	0.65
6	Magnetite MC	100	1.6	6.4	5.4	0.5	0.61	40.6	0.20	0.15	11.9	0.67
7	Magnetite MC	150	0.7	4.5	3.6	0.2	0.87	73.7	0.20	0.13	14.1	0.67
8	Magnetite MC	200	0.2	0.7	0.2	0.1	0.64	14.2	0.24	0.14	13.6	0.68

These results indicate the following:

- The feed gold grade of the two composites was extremely low (± 0.15 g/t Au or 0.0044 opt Au).
- The cleaner gravity concentrate recovered ± 15% of the gold for Endo/Exo MC and 0.7% to 6.4% of the gold for Magnetite MC.
- The silver recoveries were also low (< 9%) for the two composites.
- The higher the gravity concentrate grade for copper, the higher the gold grade.

Additional tests were performed with the high-grade Magnetite composite assaying over 2 g/t Au (0.0584 opt).

Upon completion of the initial test work, additional test work was performed with high-grade Magnetite composite, as shown on the test flowsheet shown in Figure 2. The results for this testing are summarized in Table 6.

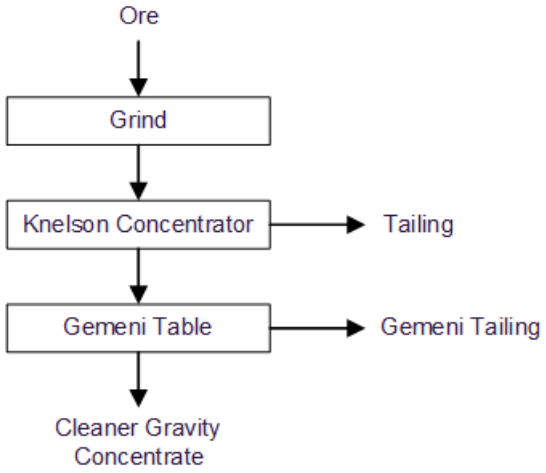


Figure 2: Gravity Testing Flowsheet for High Grade Magnetite Master Composites (Tests 9-11)

Table 6: Gravity Test Results for High Grade Magnetite Master Composite

Product	Assay			Distribution %			
	Au (g/mt)	Ag (g/mt)	% Cu	Wt	Au	Ag	Cu
Test 9: Grind P_{80} = 65 mesh							
Gemeni Conc	11.4	12.0	0.08	3.5	18.9	6.1	2.2
Gemeni Tail	4.48	<7.0	0.12	8.1	17.1	4.1	7.3
Knelson Conc	6.57	6.1	0.11	11.6	36.0	10.2	9.5
Knelson Tail	1.53	7.0	0.14	88.4	64.0	89.8	90.5
Cal Feed	2.11	6.9	0.13	100.0	100.0	100.0	100.0
Test 10: Grind P_{80} = 100 mesh							
Gemeni Conc	14.4	13.0	0.08	2.7	16.8	2.4	1.9
Gemeni Tail	3.19	15.0	0.14	8.5	11.7	8.6	10.8
Knelson Conc	5.89	14.5	0.13	11.3	28.5	10.9	12.8
Knelson Tail	1.87	15.0	0.11	88.7	71.5	89.1	87.2
Cal Feed	2.32	14.9	0.11	100.0	100.0	100.0	100.0
Test 11: Grind P_{80} = 150 mesh							
Gemeni Conc	12.5	36.0	0.09	2.4	15.9	8.1	1.6
Gemeni Tail	1.64	13.0	0.10	21.8	19.3	27.2	16.2
Knelson Conc	2.70	15.2	0.10	24.1	35.2	35.3	17.8
Knelson Tail	1.58	8.9	0.15	75.9	64.8	64.7	82.2
Cal Feed	1.85	10.4	0.13	100.0	100.0	100.0	100.0

These results indicate the following:

- The Gemeni concentrate assayed 11 to 14 g/t (0.32 to 0.408 opt) and recovered ± 16% of the gold. The low-grade of the concentrate indicates that the gold is not liberated.

The gravity test results indicate that the ore is not amenable to gravity concentration.

6. AMMONIUM THIOSULFATE LEACHING

Konnex wanted an environmentally friendly lixiviant for precious metal leaching. Hence, sodium cyanide could not be used. A laboratory in Salt Lake City was successful in extracting gold using ammonium thiosulfate (ATS).

Forte, in cooperation with the SLC Laboratory, evaluated thiosulfate as an alternative to sodium cyanide for leaching of precious metals.

The test work was undertaken for both Endo/Exo and Magnetite composites. The ores were ground to varying grind sizes and leached for 8 hours in bottles at ± 40% solids. The slurry was adjusted for pH of 9.9 with ammonium hydroxide and varying amounts of ammonium thiosulfate was added prior to leaching. Kinetic samples were collected. The ATS leach recovered gold, silver, and some copper.

The test data is given in Appendix E and the results are summarized in Tables 7 – 10.

Table 7: Ammonium Thio-Sulfate Leach (ATS) at Varying Grind Sizes for Endo/Exo and Magnetite Master Composites

Test No.	Grind Size P_{80} Mesh	Extraction % (8 Hrs Leach)			Residue Grade			Calculated Head			ATS Consumption (kg/mt)
		Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu	
Endo/Exo MC											
1	65	93.7	90.9	43.1	0.03	1.4	0.235	0.47	15.0	0.413	55.6
2	100	93.9 (78.6)	84.5 (86.1)	43.0 (43.0)	0.03	2.6	0.239	0.49	16.8	0.419	42.1
3	150	94.7	65.3	42.0	0.03	7.5	0.237	0.57	21.7	0.409	48.4
4	200	92.2	59.2	42.1	0.04	8.3	0.238	0.51	20.2	0.412	46.9
Magnetite MC											
5	65	92.4	43.4	19.7	0.04	10.2	0.452	0.53	18.0	0.601	49.5
6	100	94.6 (79.3)	49.9 (46.2)	20.5 (20.5)	0.03	7.7	0.54	0.56	15.4	0.68	49.5
7	150	94.9	25.6	22.7	0.03	21.5	0.49	0.59	28.9	0.63	47.9
8	200	89.9	65.7	27.5	0.05	5.0	0.506	0.50	14.5	0.697	30.1

Note: Leach % Solids = 40. Numbers reported in parentheses are calculated recoveries based on feed and residue assays

ATS Concentration: 80 g/L

Table 8: Effect of ATS Concentration on Metal Extraction at P_{80} of 65 mesh for Endo/Exo and Magnetite Master Composites

Test No.	ATS Concentration (g/L)	Extraction % (8 hrs Leach)			Residue Grade			Calculated Head			ATS Consumption (kg/mt)
		Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu	
Endo/Exo MC											
9	40	85.9	69.4	25.7	0.03	4.2	0.2935	0.21	13.6	0.3948	38.3
10	60	94.4	85.7	37.9	0.02	2.2	0.2563	0.36	15.6	0.4129	24.9
11	80	92.4	41.1	38.1	0.03	19.7	0.2550	0.39	33.5	0.4118	34.1
Magnetite MC											
12	40	81.2	33.8	10.1	0.04	12.2	0.600	0.21	18.4	0.6300	37.0
13	60	86.0	44.2	16.9	0.06	8.7	0.549	0.43	15.5	0.6609	27.6
14	80	89.4	37.2	21.6	0.04	12.3	0.500	0.38	19.6	0.6400	25.5

Note: Leach % Solids = 40. Tests 11 and 14 were run at 50% solids.

Grind Size, P_{80} = 65 mesh

Table 9: Effect of Recycling ATS on Metal Extraction at P_{80} of 100 Mesh for Endo/Exo and Magnetite Composite

Cycle No.	Extraction %			Residue Grade			Calculated Head			ATS Consumption (kg/mt)
	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu	
Endo/Exo Composite										
1	92.7	76.3	43.0	0.04	4.9	0.2664	0.55	20.5	0.4670	43.6
2	92.1	74.6	25.4	0.03	5.0	0.2730	0.38	19.8	0.3663	42.3
3	91.1	82.3	41.2	0.03	3.7	0.2793	0.37	20.8	0.4747	48.0
4	92.1	64.3	41.3	0.03	7.3	0.2715	0.38	20.4	0.4623	31.7
5	94.6	64.4	16.4	0.02	5.9	0.2578	0.37	16.6	0.3086	26.1
6	93.0	70.4	13.0	0.02	3.6	0.2753	0.28	12.2	0.3165	20.2
Magnetite Composite										
1	89.7	41.4	20.4	0.06	13.1	0.5738	0.58	22.3	0.7221	56.4
2	89.4	25.8	20.3	0.03	27.5	0.5767	0.47	37.1	0.7237	33.0
3	80.9	48.4	18.7	0.06	9.5	0.6018	0.31	18.4	0.7407	49.9
4	91.4	41.3	20.3	0.04	10.5	0.5933	0.46	17.9	0.7448	32.6
5	70.2	31.4	4.0	0.10	10.7	0.5761	0.34	15.6	0.5999	31.7
6	91.3	45.2	7.4	0.04	6.6	0.6048	0.46	12.0	0.6531	17.8

Note: Leach Time: 8 Hours

Table 10: ATS Testing of Variability in Flotation Tailing Samples

Leach Test	Flotation Test No.	Leach Feed Grade			Extraction %			Leach Residue Grade			ATS Consumption (kg/mt)
		Au (g/mt)	Ag (g/mt)	% Cu	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	
27	17	0.08	11.3	0.21	89.7	68.5	51.0	0.06	4.3	0.195	51.6
37	20	0.12	10.0	0.43	69.8	54.7	24.8	0.11	<7	0.326	50.2
38	21	0.49	7.1	0.31	36.6	41.6	25.1	0.47	<7	0.239	48.8
39	22	0.38	13.0	0.14	49.5	34.1	13.0	0.35	11.0	0.121	52.0
40	23	0.07	8.5	0.29	77.2	28.4	7.1	0.08	<7	0.259	54.5
41	24	0.10	40.0	0.35	78.1	15.2	21.0	0.09	36.6	0.320	45.9
42	25	0.17	10.0	0.44	65.8	20.7	17.5	0.16	9.3	0.343	44.6
43	26	0.10	5.0	0.22	74.1	27.9	14.7	0.10	<7	0.195	50.5
44	27	0.07	8.0	0.12	34.6	18.4	20.7	0.49	<7	0.097	39.0
46	LCT1-T	-	-	-	72.7	28.3	33.6	0.10	16.6	0.206	54.3
48	LCT2-T	-	-	-	70.8	22.6	13.3	0.12	20.0	0.659	62.0

These results indicate the following:

- A majority of the gold and some silver and copper are extracted using ATS leach.
- The ammonium hydroxide and ammonium thiosulfate consumption were high. For example, approximately 100 kg/mt of ammonium hydroxide and 30 to 55 kg/mt of ATS were consumed to extract the precious metals.
- The calculated head for gold was significantly higher than assayed head (± 0.5 g/t vs. 0.15 g/t Au). Hence, the gold extraction based on feed and residue assays were lower than shown in the tables. This is due to interference of ammonium in the analytical procedures. The recoveries were calculated based on feed and residue assays. The biggest difference was for gold. Approximately 15% lower recovery was calculated as compared to recovery based on solution and residue assays.
- Locked-cycle testing of recycling ATS demonstrated that it has no effect on leach kinetics up to six cycles (Table 9).
- ATS testing of variability samples clearly demonstrate that the extractions based on solution analyses and residue are erroneous (Table 10). Leach test No. 44 had feed assaying 0.07 g/t Au and the residue assaying 0.49 g/t Au.

7. SULFURIC ACID LEACHING

The mineralogy of the composite samples from earlier test work has indicated that copper minerals present in the ore are secondary copper (chalcocite/covellite), oxide copper, and copper silicates. The flotation process can recover secondary copper and some oxide copper following the sulfidization process but will not recover copper silicates. These are readily leachable with sulfuric acid.

Flotation tailing from varying grinds, variability samples, and locked-cycle tests were subjected to sulfuric acid leaching. The acid concentration was maintained at 20 g/L. The tests were originally run for 24 hours but later tests were performed for only 8 hours since most of the copper is extracted in ± 6 hours.

The test data is given in Appendix F and the results are summarized in Table 11 and Table 12.

Table 11: Summary of Acid Leach Test on Flotation Tailings at Varying Grind Sizes

Test No.	Flotation Test No.	Grind P_{80} Mesh	Extraction % Cu		Leach Residue % Cu	Calculated Head % Cu	Acid Consumption (kg/mt)	
			8 Hrs	24 Hrs			8 Hrs	24 Hrs
Endo/Exo MC								
15	9	65	72.4	78.9	0.059	0.2799	53.47	71.41
16	10	100	77.3	80.5	0.049	0.2526	55.18	72.93
17	11	150	80.4	80.7	0.050	0.2563	57.32	75.21
18	12	200	72.6	82.3	0.049	0.2763	61.11	79.59
Magnetite MC								
19	13	65	-	55.5	0.2576	0.5793	-	38.93
20	14	100	51.6	58.9	0.2664	0.6487	31.10	38.87
21	15	150	60.2	58	0.2703	0.6441	30.84	38.24
22	16	200	56.1	63.8	0.2775	0.7660	36.74	45.17

Note: Sulfuric Acid addition was 20 g/L

Table 12: Summary of Acid Leach Test on Flotation Tailings for Variability Samples

Test No.	Tailing Flotation Test	Ore Type	Extraction % Cu		Calculated Head % Cu	Acid Consumption (kg/mt)	
			8 Hrs	24 Hrs		8 Hrs	24 Hrs
28	17	Endo/Exo	80.8	82.2	0.308	57.30	74.85
29	20	Exo	75.6	-	0.468	37.96	-
30	21	Exo	76.5	-	0.3606	20.97	-
31	22	Endo	39.3	-	0.152	39.3	-
32	23	Endo	43.1	-	0.320	25.1	-
33	24	Endo	64.4	-	0.466	51.8	-
34	25	Endo/Exo	74.9	-	0.541	42.8	-
35	26	Endo	62.4	-	0.295	27.0	-
36	27	Mag HG	45.2	-	0.123	14.56	-
45	LCT-1	Endo/Exo MC	84.3	-	0.392	61.2	-
46	LCT-2	Mag MC	62.3	-	0.789	36.5	-

Note: Sulfuric Acid addition was 20 g/L

These results indicate the following:

- A majority of the copper in the Endo/Exo MC flotation tailing is leached in 8 hours. The leached residue assayed ± 0.05% Cu. The acid consumption was ± 75 kg/mt.
- The extraction of copper from Magnetite MC flotation tailing was low (± 55%). However, the acid consumption was ± 40 kg/mt.
- The variability leach results indicate that 8 hours of leach consumes significantly less acid (57.3 kg/t) as compared to 24 hours of leach (75 kg/t). The additional copper extraction was only 1.4%.

These results indicate that sulfuric acid will extract copper which did not float (i.e. copper silicates).

8. FLOTATION TESTING

The rougher and cleaner flotation test data is given in Appendix G. The locked-cycle test data is given in Appendix H.

8.1 Rougher Flotation Tests

Rougher flotation tests were performed with the two composites to determine the amenability of recovering gold, silver, and copper. The tests were performed at varying grind sizes with and without sulfidization of the oxide minerals. The reagent suite consisted of potassium amyl xanthate, Aero promotors 208 and 404, and methyl isobutyl carbinol (MIBC) as a frother. NaHS was used as a sulfidizing agent.

The results are summarized in Table 13 and Table 14.

Table 13: Summary of Flotation Test Results for Endo/Exo and Magnetite Master Composites for Varying Grind Sizes

Test No.	Grind P_{80} Mesh	Recovery %				Concentrate Grade			Calculated Feed		
		Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
Endo/Exo Composite											
1	65	2.6	40.4	28.6	21.7	2.24	196.4	3.50	0.15	18.1	0.43
2	100	4	45.3	36.2	23.0	1.58	156.0	2.46	0.14	17.3	0.43
3	150	3.6	44.3	32.1	24.9	1.53	188.1	2.99	0.12	21.4	0.44
4	200	5.8	35.6	26.9	21.2	1.04	90.3	1.50	0.14	19.5	0.41
Magnetite Composite											
5	65	10.2	32.6	12.4	12.6	0.49	19.8	0.80	0.14	16.4	0.65
6	100	7.6	38.6	22.3	9.2	0.59	31.0	0.82	0.14	10.6	0.68
7	150	11.1	42.5	21.9	12.4	0.46	30.0	0.77	0.13	15.1	0.69
8	200	14.5	47.3	25.5	17.4	0.39	26.1	0.86	0.13	14.8	0.72

Note: Flotation Time: 9 Minutes

Table 14: Summary of Flotation Test Results for Endo/Exo and Magnetite Master Composites for Varying Grind Sizes and Sulfidization

Test No.	Grind P_{80} Mesh	Recovery %				Concentrate Grade			Calculated Feed		
		Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
Endo/Exo Composite											
9	65	3.6	47.3	60.5	45.6	1.67	280.4	5.91	0.13	16.8	0.47
10	100	2.9	51.8	38.5	45.4	2.48	393.8	7.04	0.14	30.1	0.46
11	150	2.9	44.0	66.3	46.4	2.11	422.0	7.73	0.14	18.4	0.48
12	200	3.3	47.1	76.2	44.6	2.07	429.8	6.0	0.15	18.8	0.45
Magnetite Composite											
13	65	7.5	41.4	27.0	14.0	0.88	62.9	1.25	0.16	17.4	0.67
14	100	8.8	38.5	30.4	15.7	0.65	42.5	1.20	0.15	12.3	0.67
15	150	15.4	47.3	48.1	24.6	0.39	38.2	1.05	0.13	12.2	0.66
16	200	11.1	47.1	42.9	19.9	0.64	48.0	1.16	0.15	12.4	0.65

Note: Flotation Time: 9 Minutes
Sulfidization with 500 g/mt NaHS

These results indicate the following:

- The rougher concentrate recovered 35% to 45% of gold, 26% to 36% of silver, and 21% to 25% of copper from Endo/Exo MC. The concentrate assayed 1 to 2.2 g/t Au, 90 to 196 g/t Ag, and 1.5% to 3.5% Cu.
- The rougher concentrate for the Magnetite MC recovered 33% to 47% of gold, 12% to 25% of silver, and 9% to 17% of copper. The concentrate grades were lower because a significantly higher proportion of feed floated (i.e. ± 10% as compared to ± 4% for Endo/Exo MC).
- Sulfidization of the oxide minerals significantly improved both recovery and grade of silver and copper in the rougher concentrate. The concentrate from Endo/Exo MC assayed 1.67 to 2.48 g/t Au, 280 to 430 g/t Ag, and 6% to 7.7% Cu. The metal recoveries in the concentrate were 44% to 52% of gold, 38.5% to 76.2% of silver, and ± 46% of copper.

8.2 Cleaner Flotation Tests

Two 10-kg rougher flotation tests were performed to generate a concentrate for cleaner flotation tests with the primary objective of determining if marketable-grade copper concentrate can be produced. The rougher flotation results are summarized in Table 15, and the cleaner flotation results using the process flowsheet shown in Figure 3 are summarized in Table 16.

Table 15: Rougher Flotation Test at P_{80} of 100 Mesh in One Cubic Foot Flotation Cell for Endo/Exo Master Composite (Test No. 17)

Product	Distribution %				Assays		
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu
Rougher Concentrate	1.3	51.4	46.9	39.3	6.05	762	10.47
Rougher Tailing	98.7	48.6	53.1	60.7	0.08	11.3	0.21
Calculated Feed	100	100	100	100	0.15	21.0	0.345

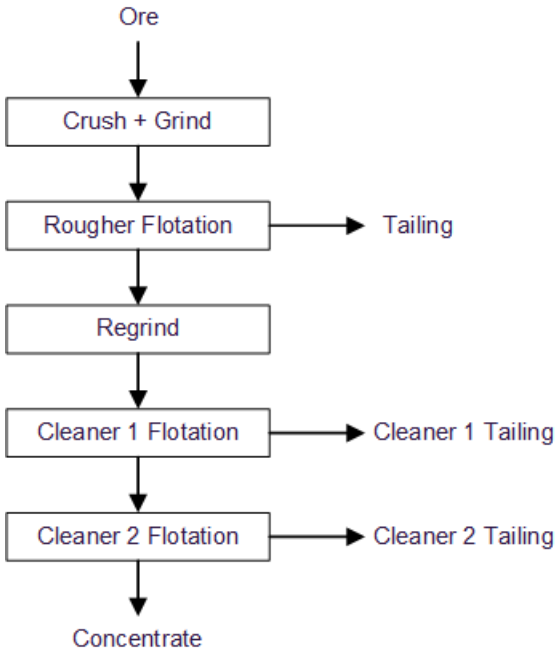


Figure 3: Rougher and Cleaner Open-Circuit Test Flowsheet

Table 16: Cleaner Flotation Tests With/Without Regrind of Rougher Concentrate from Flotation Test No. 17

Product	Distribution %				Assays		
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu
Test No. 18 (No Regrind)							
Cleaner 2 Conc	23.0	53.8	70.0	74.8	16.0	2330	34.14
Cleaner 2 Tail	10.3	7.4	7.5	5.8	4.92	557	5.93
Calc Cleaner 1 Conc	33.4	61.3	77.5	80.7	12.57	1780.6	25.40
Cleaner 1 Tail	66.6	38.7	22.5	19.3	3.98	259	3.05
Calc Rougher Conc	100	100	100	100	6.85	767	10.51
Test No. 19 (2.5 min Regrind)							
Cleaner 2 Conc	16.6	51.0	56.2	62.2	16.10	2560	39.05
Cleaner 2 Tail	7.1	9.0	5.3	4.9	6.63	563	7.22
Calc Cleaner 1 Conc	23.7	60.0	61.5	67.1	13.26	1961.6	29.51
Cleaner 1 Tail	76.3	40.0	38.5	32.9	2.75	382	4.50
Calc Rougher Conc	100	100	100	100	5.24	756	10.43

These results indicate the following:

- One-cubic-foot flotation cell produced similar recoveries as the small flotation cell but recovered a higher grade concentrate. The overall weight recovery was 1.3%.
- The cleaner flotation tests indicated that a marketable-grade copper concentrate can be produced from Endo/Exo MC with one cleaner-flotation stage and no regrind required. The concentrate assayed 12.57 g/mt Au, 1780.6 g/mt Ag, and 25.40% Cu.

8.3 Locked-Cycle Tests

A locked-cycle flotation test was performed to simulate the flotation process for recovery of copper and precious metals in a plant.

A six-cycle locked-cycle test employed the process flowsheet shown in Figure 4. The test data for the two composites is given in Appendix H and the results are summarized in Table 17.

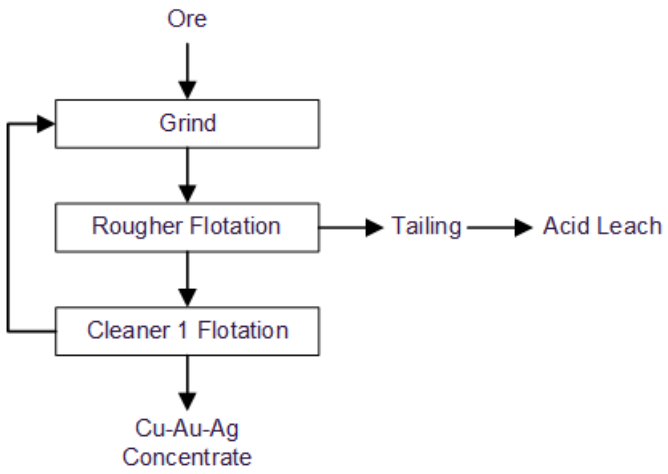


Figure 4: Locked-Cycle Test Flowsheet

Table 17: Summary of Locked-Cycle Test Results

Product	Distribution %				Assays		
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu
Endo/Exo MC							
1st Cl Conc	0.4	39.3	36.2	33.5	17.43	2288	35.7
1st Cl Tail	0.1	0.8	0.5	0.3	2.184	203	1.8
Calc Ro Conc	0.5	40.1	36.7	33.8	14.38	1871	28.92
Ro Tail	99.5	59.9	62.3	66.2	0.11	15.94	0.282
Calc Feed	100.00	100.0	100.0	100.0	0.18	25.3	0.43
Magnetite MC							
1st Cl Conc	0.4	20.9	11.6	3.8	10.68	563	6.0
1st Cl Tail	0.1	0.3	0.2	0.2	0.628	47	1.0
Calc Ro Conc	0.5	21.2	11.8	4.0	8.67	459.8	5.0
Ro Tail	99.5	78.8	88.2	96.0	0.15	16.29	0.572
Calc Feed	100.0	100.0	100.0	100.0	0.19	18.5	0.60

These results indicate the following:

- The cleaner 1 flotation concentrate, assaying 35.7% Cu, 17.43 g/t Au (0.5090 opt), and 2288 g/t Ag (66.81 opt), recovered 33.5% of copper, 39.2% of gold, and 36.2% of silver in 0.4% of the feed weight for Endo/Exo MC.
- The Magnetite MC did not produce a marketable-grade copper product. The cleaner 1 concentrate assayed only 6% Cu.

It is important to note that one will not process 100% Magnetite material in the plant. Additional locked-cycle tests should be run with a blend of Endo/Exo and Magnetite material based on the mine plan.

8.4 Open-Cycle Rougher and Cleaner Flotation Tests for Variable Feed Materials

Open-cycle rougher and cleaner flotation tests were performed with the variability samples. The test data are given in Appendix G and the results are summarized in Table 18 and Table 19.

Table 18: Flotation Test Results for Variability Samples

Test No.	Ore Type	Sample Description	Rougher Recovery %				Tailing Grade			Calculated Feed		
			Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
17	Endo/Exo	MC	1.3	51.4	46.9	39.3	0.08	11.3	0.21	0.15	21.0	0.345
20	Exo	102	3.3	58.6	45.0	11.4	0.12	10.0	0.43	0.35	19.9	0.50
21	Exo	101/103	1.4	50.1	20.3	10.2	0.49	7.1	0.31	0.97	8.8	0.34
22	Endo	298/667	4.9	62.6	33.4	70.6	0.38	13.0	0.14	0.97	18.5	0.47
23	Endo	104/596	1.8	25.7	3.5	4.8	0.07	8.5	0.29	0.093	8.6	0.30
24	Endo	291	4.6	14.8	4.3	7.3	0.10	40.0	0.35	0.11	40.0	0.36
25	Endo/Exo	679/14	5.1	48.1	31.7	12.9	0.17	10.0	0.44	0.31	13.9	0.48
26	Endo	127/128	1.3	45.3	12.6	13.7	0.10	5.0	0.22	0.18	5.6	0.25
27	Mag	High Grade	0.7	73.8	8.1	3.9	0.74	8.0	0.12	2.81	8.6	0.12

Note: 9 minutes of rougher flotation at P_{80} = 100 mesh

Table 19: Cleaner Flotation Concentrate Grades for Variability Samples

Test No.	Ore Type	Sample Description	Calculated Feed Grade			1st Cleaner Concentrate Grade		
			Au (g/mt)	Ag (g/mt)	% Cu	Au (g/mt)	Ag (g/mt)	% Cu
17	Endo/Exo	MC	0.15	21.0	0.345	-	-	-
20	Exo	102	0.35	19.9	0.50	17.9	802	2.04
21	Exo	101/103	0.97	8.8	0.34	13.7	381	6.22
22	Endo	298/667	0.97	18.5	0.47	31.8	307	17.43
23	Endo	104/596	0.093	8.6	0.30	4.01	25.0	1.07
24	Endo	291	0.11	40.0	0.36	0.76	67.0	0.58
25	Endo/Exo	679/14	0.31	13.9	0.48	14.2	398	2.81
26	Endo	127/128	0.18	5.6	0.25	18.4	145	6.95
27	Mag	High Grade	2.81	8.6	0.12	1129	350	1.61

Note: 9 minutes of rougher flotation at P_{80} = 100 mesh

6 minutes of cleaner flotation

These results indicate the following:

- There appears to be no correlation between copper recovery in the flotation process and feed grade. However, the overall recovery of copper in the combined flotation and leach process is over 90% (Figure 5). A few samples did show lower copper recovery.
- The grade of the cleaner 1 concentrate (Table 19) indicated three of eight samples had the potential to produce ± 25% copper concentrate. The grades for cleaner 1 ranged from 6.22 to 17.43% Cu. It appears that two cleaner stages may be needed to achieve ± 25% copper in the concentrate.
- Both copper recovery and concentrate grade for the Magnetite composite and high-grade Magnetite composite were not good.
- The higher the gold feed grade, the higher the gold recovery in the flotation process (Figure 6).
- The silver in the copper concentrate will help in marketing lower grade copper concentrate.

More flotation testing is recommended to ensure that the flotation concentrate can be marketed.

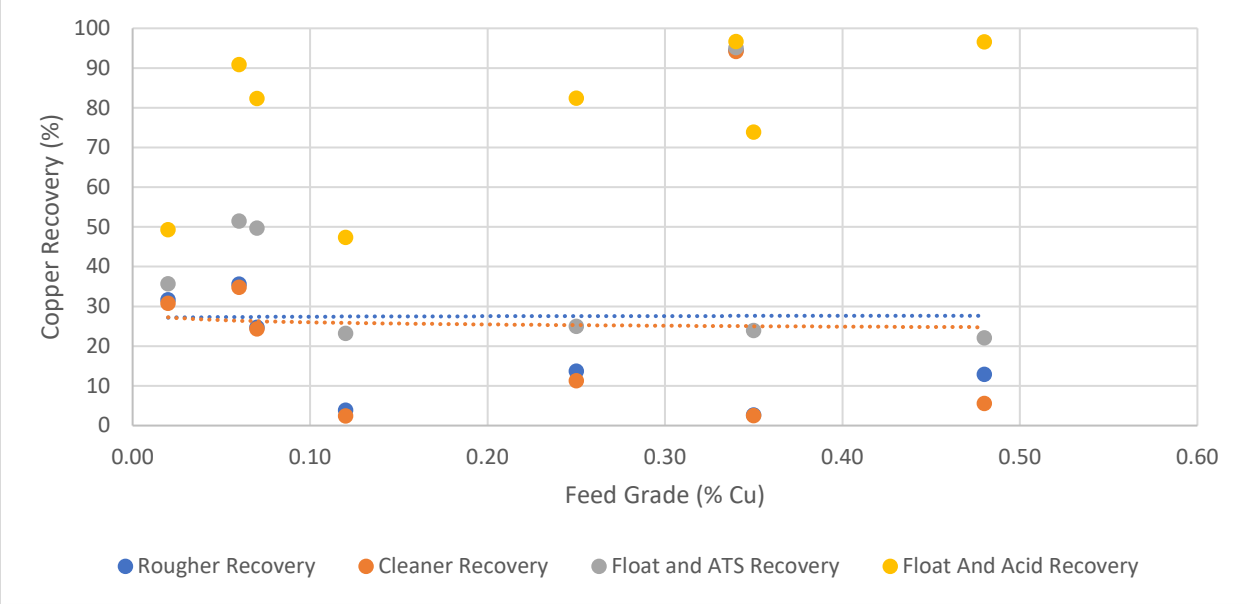


Figure 5: Copper Recovery as Function of Feed Grade

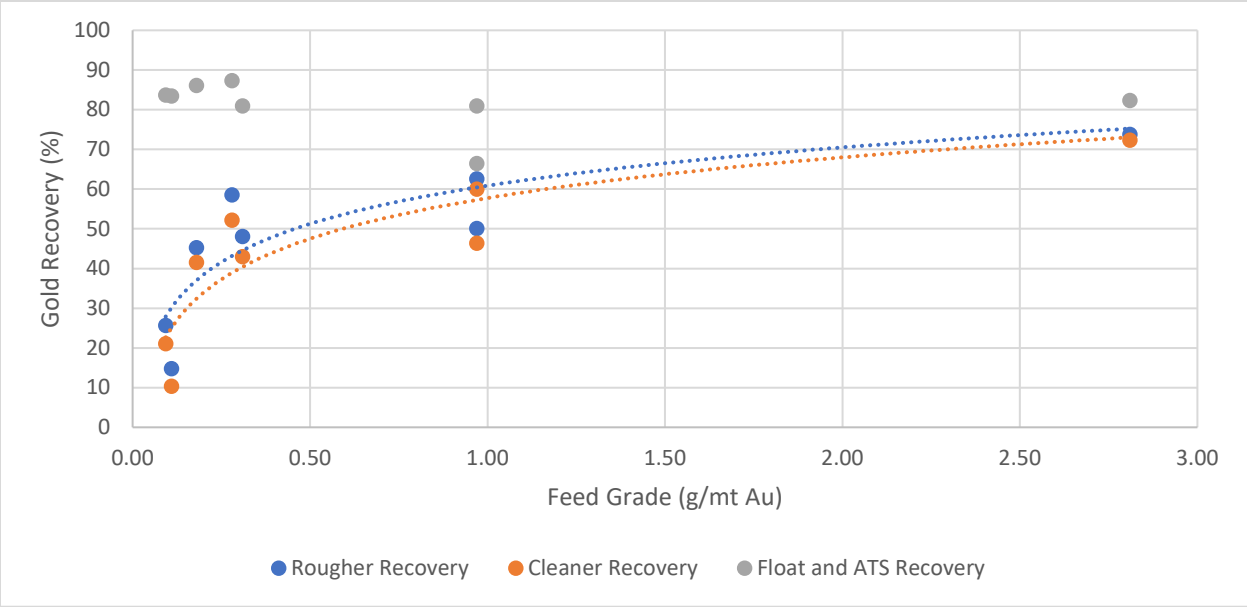


Figure 6: Gold Recovery as Function of Feed Grade

9. SMELTER PENALTY ANALYSES OF FLOTATION CONCENTRATES

The cleaner 1 concentrates from the two locked-cycle tests were submitted for smelter penalty analyses. The concentrate from the Endo/Exoskarn master composite assayed 35.3% Cu (LCT-1), whereas the concentrate from the Magnetite master composite assayed only 6% Cu (LCT-2). The smelter penalty and chemical analyses of the concentrate are given in Appendix I and the results are summarized in Table 20 and Table 21.

Table 20: Smelter Penalty Analyses of Copper Concentrates

Element	Endo/Exo Conc (LCT-1)	Magnetite Conc (LCT-2)
As, ppm	2937	841
Bi, ppm	3544	398
Sb, ppm	47	138
Hg, ppm	16.70	66.10
Se, ppm	298	33
Te, ppm	388	88
SiO_2, %	20.07	41.29
MgO, %	2.25	2.00
P_2O_5, %	0.41	0.83
Chloride, ppm	19,600	104
Fluoride, ppm	204	500

Table 21: Chemical Penalty Analyses of Copper Concentrates

Element %	Copper Concentrate	
	Endo/Exo MC (LCT-1)	Magnetite MC (LCT-2)
Al	0.93	4.70
Ca	2.28	3.01
Cu	34.61	6.52
Fe	13.81	16.61
K	0.32	0.56
Mg	1.30	1.21
Mn	0.17	1.31
Mo	0.16	0.07
Na	0.16	0.07
Pb	0.26	0.11
Ti	0.04	0.20
Zn	0.40	0.83
ppm		
Ag	1936	514
Ba	244	751
Cd	43	33
Co	132	140
Cr	43	92
Ni	165	109
Sr	48	117
V	37	68
W	497	0.83

The deleterious elements in the concentrate are arsenic, antimony, cadmium, selenium, bismuth, and tellurium. Limits are also imposed on mercury, lead, zinc, and nickel.

Additional testing needs to be undertaken to determine the level of the impurities in the copper concentrate which will assist in marketing the concentrate.

10. THICKENING TEST

Thickening tests were performed on flotation tailings for both Endo/Exo and Magnetite master composites. These samples were generated at a primary grind of P_{80} of 100 mesh. A high weight anionic polymer (DAF-10) manufactured by Polymer Ventures was used at 17 g/mt to thicken the solids. Additional tests were also performed following the acid leaching of the flotation tailings. The test data is given in Appendix J and the results are summarized in Table 22.

Table 22: Thickening Test Results

Sample	P_{80} µm	pH	Flocculant	Feed % Solids	Unit Area Required (ft^2/t/day)		
					45%	50%	55%
Endo/Exo Flotation Tailing	150	7.9	17 g/t DAF-10	25	0.015	0.038	0.33
Mag MC Flotation Tailing	150	7.9	17 g/t DAF-10	25	0.035	0.089	6.384
Endo/Exo Flotation Tail Acid Leach Residue	150	3.0	17 g/t DAF-10	16	0.799	2.829	-
Mag MC Flotation Tailing Acid Leach Residue	150	3.0	17 g/t DAF-10	15	0.202	0.498	-

These results indicate the following:

- Flotation tailings can be thickened to 55% solids for the Endo/Exo MC using 17 g/mt of flocculant DAF-10 or any other similar flocculant. Unit area required was calculated to be 0.33 ft^2/t/day.
- Magnetite MC flotation tailings would require 6.384 ft^2/t/day to thicken to 55% solids. However, one would not be processing this material separately.
- The thickener area requirements for acid leach residues was high and the thickener diameter would be very large.

The acid leach residue would be directly sent to belt filter where the pregnant solution can be separated and the residue can be easily washed.

11. FILTRATION TESTS

Filtration tests were performed on acid leach residue for both Endo/Exo and Magnetite composites. Both vacuum and pressure filtration tests were performed. The test data is given in Appendix K and the results are summarized in Table 23.

Table 23: Summary of Vacuum and Pressure Filtration Results

Sample	Feed % Solids	Vacuum in Hg	Pressure psi	% Cake Moisture	Dry lbs/ft^2/hr
Endo/Exo MC Leach Residue	38	16.5	-	24.5	128.4
Magnetite MC Leach Residue	39	16.5	-	25.9	95.2
Endo/Exo MC Leach Residue	41	-	80	16.2	911.3
Endo/Exo MC Leach Residue	40	-	80	15.7	1103.8
Magnetite MC Leach Residue	39	-	80	16.6	574.7
Magnetite MC Leach Residue	39	-	80	17.2	272.3

Note: Grind P_{80} = 100 mesh

Feed: Flotation Tailing

These results indicate the following:

- Vacuum filtration rates were significantly lower than pressure filtration rates.

Additional studies with a mixture of the two ore types are recommended in order to accurately size the belt filter.

12. CEMENTATION TESTING

A preliminary cementation test was performed on the pregnant solution from acid leaching of Endo/Exo master composite flotation tailings. The data is given in Appendix L and the results are summarized in Table 24 and Table 25.

Table 24: Solution Analyses of Pre- and Post-Cementation for Endo/Exo Leach Test

Element, ppm	Pre-Cementation (Preg Solution)	Post-Cementation (Barren Solution)
Al	1158	1244
Ca	498	631
Cd	3.58	3.11
Co	8.22	8.98
Cu	1608	90.6
Fe	3542	6211
P	364	25.9

Table 25: Analyses of Cemented Copper

Element %	Cemented Copper
Cu	28.7
Fe	41.2
P	2.34
S	1.01
Zn	0.04

The cementation test was performed for a residence time of 2.5 hours. Approximately 94.4% of the copper was precipitated. Due to excess residence time, a significant amount of iron fillings were dissolved. The cemented copper assayed 28.7% Cu and 41.2% Fe.

Additional testing is needed to optimize the process parameters.

13. EVALUATION OF ALTERNATIVE PROCESSING FLOWSHEETS

Due to the mineralogy of the copper minerals in the deposit (i.e. sulfide/oxide/silicates), the low grade of the ore, and the need for non-cyanide lixiviant, several alternative flowsheets were evaluated in order to select the optimum process.

13.1 Gravity/Flotation Process

The process flowsheet is shown in Figure 7. The test results for the gravity and flotation process are summarized in Table 26 at 100 mesh for the Endo/Exo and Magnetite master composites. These test results were combined from independent tests performed by gravity and flotation on feed and should be considered as maximum recoveries at these process conditions.

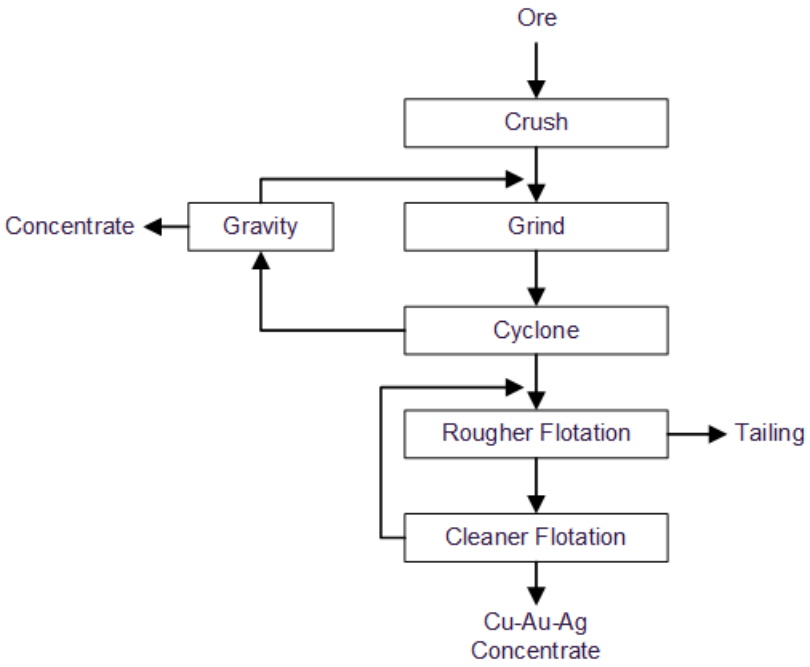


Figure 7: Simplified Gravity/Flotation Flowsheet

Table 26: Gravity and Flotation Results for the Two Master Composites at P_{80} = 100 mesh

Test No.	Recovery %				Grade		
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu
Endo/Exo MC							
EMGT-2	0.6	10.8	3.9	3.8	2.75	156.0	2.83
EMFT-10	2.9	51.8	38.5	45.4	2.48	393.8	7.04
Combined	3.5	62.6	42.4	49.2	2.53	353.0	6.32
Magnetite MC							
EMGT-6	1.6	6.4	5.4	0.5	0.61	40.6	0.20
EMFT-14	8.8	38.5	30.4	15.7	0.65	42.5	1.20
Combined	10.4	44.9	35.8	16.2	0.64	42.2	1.05

These results indicate the following:

- The gravity concentration process recovered a maximum of 10.8% Au for Endo/Exo MC. The gravity recoveries may be low because of the feed assaying only 0.14 g/t Au.
- The flotation process recovered a reasonable amount of gold (38.5% to 51.8%), silver (30.4% to 38.5%), and copper (45.7% to 45.4%).
- The combined gravity and rougher flotation process recovered 62.6% Au, 42.4% Ag, and 49.2% Cu. The concentrate grade assayed 2.53 g/t Au, 353 g/t Ag, and 6.32% Cu for the Endo/Exo MC.
- The copper recovery for the Magnetite MC was only 16.2% in the combined gravity and flotation concentrate. The concentrate grade was only 1.05% Cu.

13.2 Agitated ATS and Acid Leach Process

The simplified process flowsheet for the agitated leach circuit is shown in Figure 8. The ground ore will be first leached with ATS, followed by separation of pregnant solution from solids. The pregnant solution will

be sent to the electrowinning (EW) circuit where gold, silver, and some copper will precipitate. The solids will be leached with sulfuric acid to recover copper.

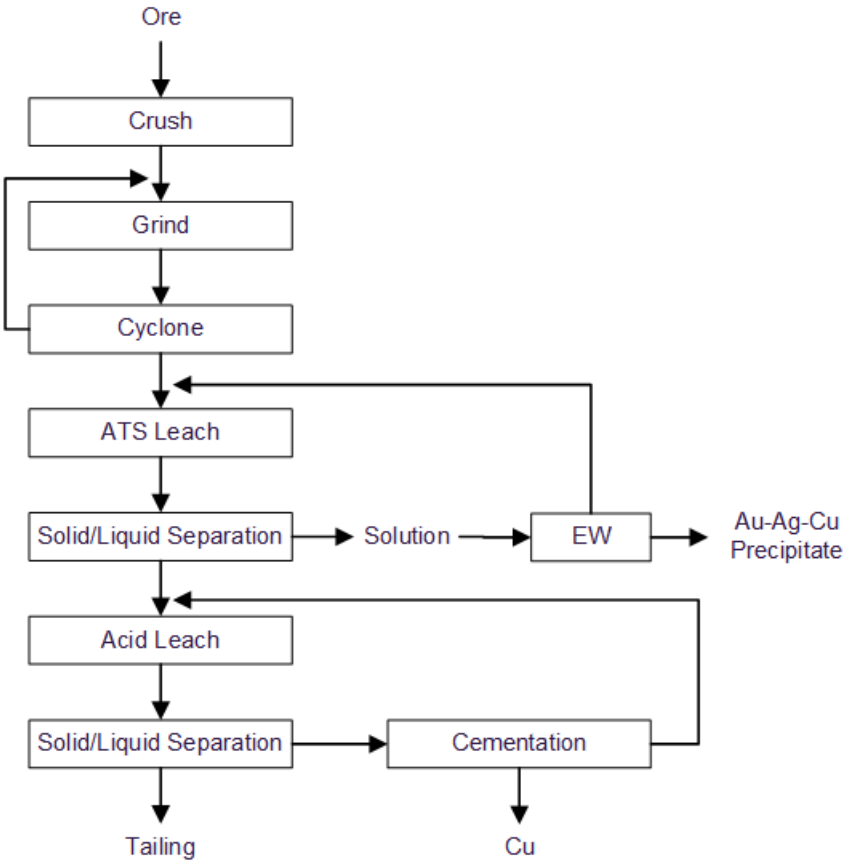


Figure 8: Simplified ATS/Acid Leach Flowsheets

Assuming copper that floated was recovered by ATS process, material balance was calculated for both composites at P_{80} of 100 mesh. The results are summarized in Table 27.

Table 27: Combined Results for Two-Stage Leach Process at P_{80} = 100 mesh

Test No.	Recovery %			Consumption (Reagent kg/t)
	Au	Ag	Cu	
Endo/Exo MC				
EMBR-2	78.6	86.1	43	ATS 42.1
EMBR-16	-	-	41	H_2SO_4 55.0
Total	78.6	86.1	84	
Magnetite MC				
EMBR-6	79.3	46.2	20.5	ATS 49.5
EMBR-20	-	-	41	H_2SO_4 31.1
Total	79.3	46.2	61.5	

Note: 8 hrs leach time for acid

These results indicate the following:

- The recovery of gold, silver, and copper in the two-stage leach process, excluding losses in electrowinning circuit, was 78.6% Au, 86.1% Ag, and 84% Cu for Endo/Exo MC. The recovery from Magnetite MC was 79.3% Au, 46.2% Ag, and 61.5% Cu.
- The reagent consumptions (ATS and acid) were high (31 kg/t to 55 kg/t). Based on the price of $270/mt for ATS, the reagent cost would be $13.50/mt for ATS. Assuming the feed assays 0.5 g/t Au, the incremental recovery using ATS versus flotation would be ± 25% (or △R = $7.50 based on 60 g/t Au).
- The acid consumption of 50 kg/t based on a price of $150/mt equates to $7.50. The incremental copper recovery from flotation tailing would be 55% (90% - 35% based on the locked-cycle test). This is equivalent to $19.25/mt based on $3.50/lb of copper.

These results indicate that the flotation tailing should be subjected to acid leach for recovery of copper instead of ATS leach for recovery of gold.

The assay-by-size test data for flotation tailing is given in Appendix M. The data indicated that both gold and copper were equally present in all size fractions. Hence, one could not reduce the tonnage to the leach circuit.

13.3 Flotation/Acid Leach of Tailing

The simplified flowsheet for flotation followed by leaching of the flotation tailing is shown in Figure 9. Both ATS and acid leach tests were performed on the flotation tailing. However, acid leaching to recover copper was more economical than recovery of gold. Hence, the flowsheet shows only acid leaching.

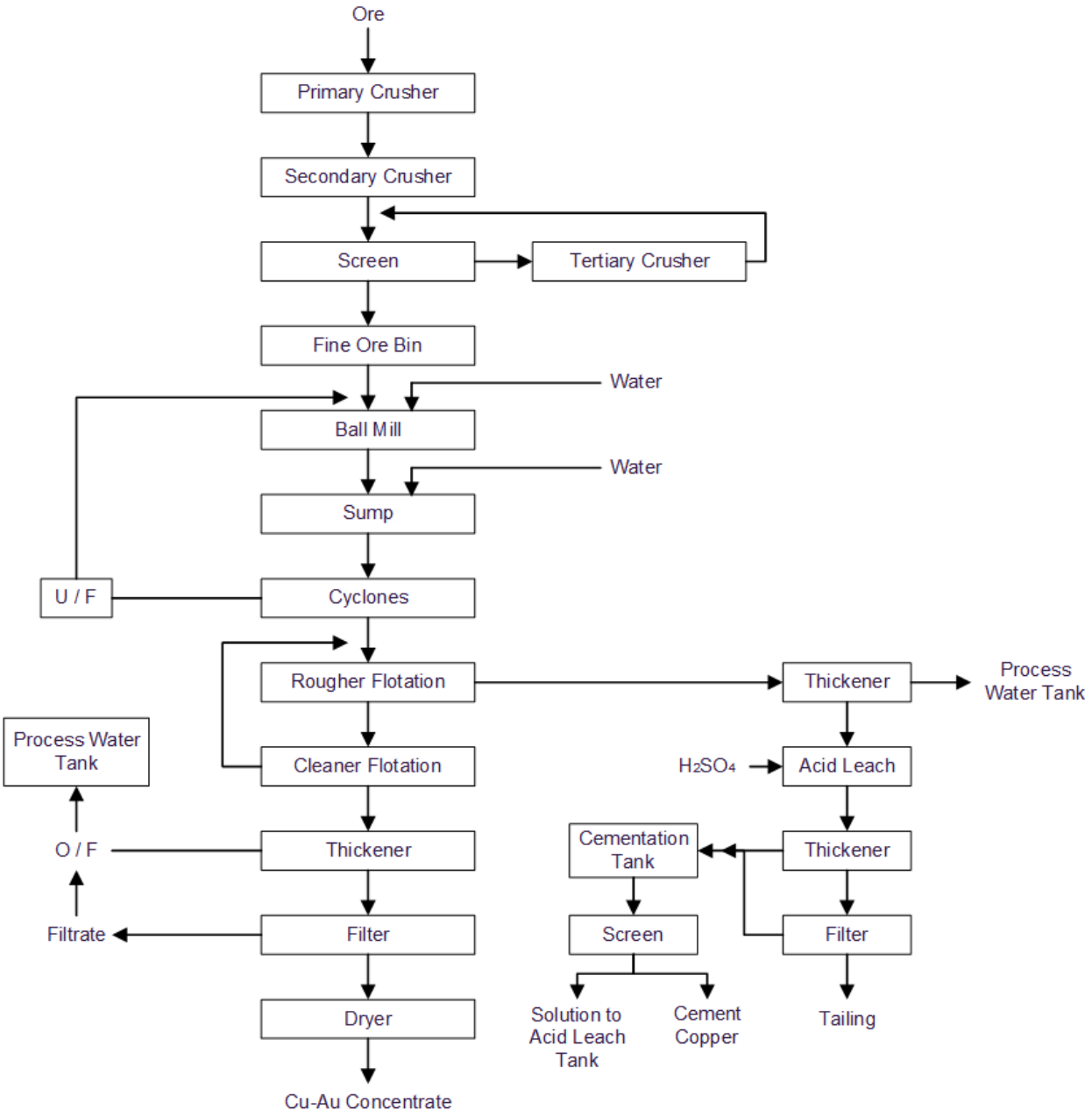


Figure 9: Simplified Flotation/Acid Leach Process Flowsheet

The material balance for the flowsheet is presented in Table 28. The balance is based on one locked-cycle flotation test on Endo/Exo MC and an assumption that the feed grade would be 0.5 g/t Au, 18.7 g/t Ag, and 0.5% Cu.

Table 28: Metal Recoveries for Flotation/Acid Leach Process for Endo/Exo Master Composite

Test No.	Recovery %				Concentrate Grade			Acid Consumption (kg/mt)
	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	% Cu	
LCT-1 Cl1 Conc	0.4	50.0	36.2	33.5	17.43	2288	35.7	
45 Acid Leach	-	-	-	56.1	-	-	-	61.2
Total	0.4	50.0	36.2	89.6	-	-	-	

The overall recoveries of gold, silver, and copper would be 50% Au, 36.2% Ag, and 89.6% Cu.

14. CONCLUSIONS

The following conclusions can be drawn based on the metallurgical test work:

- Endoskarn and Exoskarn represent the majority of the oxide ore in the prospect. Magnetite constitutes a small portion of the prospect.
- The metallurgical study was undertaken on two master composites – Endo/Exoskarn and Magnetite ore.
- The Endo/Exo composite assayed 0.14 g/t Au, 18.7 g/t Ag, and 0.43% Cu. The Magnetite composite assayed 0.145 g/t Au, 14.3 g/t Ag, and 0.75% Cu.
- Both composites were slightly abrasive and exhibited average hardness.
- Both composites were not amenable to gravity concentration due to low-grade.
- A marketable-grade copper concentrate could not be produced from the Magnetite composite.
- The copper recovery in the acid leaching of magnetite ore was also low as compared to the Endo/Exo composite.
- Several processing options were evaluated, including gravity/flotation, two-stage leach (ATS and acid), and flotation/acid leach processes. The latter process was the most techno-economically viable.

15. RECOMMENDATIONS

The following recommendations are made based on the metallurgical studies undertaken in this phase of the test program:

- Composites representing the first three years of operation should be prepared and tested in the flowsheet developed for the deposit.
- Additional thickening, filtration, and cementation tests should be undertaken to properly size the equipment.
- A marketing study should be undertaken to find the buyer for the flotation concentrate and the cemented copper.

APPENDIX A: PREPARATION OF COMPOSITE SAMPLES

Re: Empire Composites

Ryan Allen <Ryan@RDIminerals.com>

Fri 3/24/2023 11:44 AM

To:Deepak Malhotra <Deepak@RDIminerals.com>

Hello Deepak,

I have taken the lithology data from Jessica and calculated the amount of each ore type and expected grades for each hole. The data is summarized in the tables below. Jessica stated that the purpose of this drilling was to acquire the Febx lithology and she would like to keep it separate if possible. There is plenty of it and the average Cu grade is near the deposit average. I would expect to old this out of the two master composites and test as a variability sample at the end of the program.

The Endo/Exo master composite should be fine. The Mag material is only 59 kgs and is significantly higher grade than the deposit average. How would you like to handle this? Should some of the Febx be mixed in? Let me know your thoughts.

Thanks,

Ryan

	Lithology	Mass (kg)	Au (g/mt)	Ag (g/mt)	Cu (%)
Hole 1	ENDO/EXO/QFP	143	0.01	12.1	0.59
	Mag	10	0.01	42.9	2.49
	Febx	3	0.01	26.7	0.33
Hole 2	ENDO/EXO/QFP	150	0.02	3.2	0.06
	Mag	21	0.02	4.4	0.11
	Febx	34	0.01	5.1	0.09
Hole 3	ENDO/EXO	63	0.24	5.6	0.29
	Mag	28	0.37	19.1	2.66
	Febx	37	0.21	18.8	0.76
Hole 4	ENDO/GP	171	0.04	12.5	0.52
Hole 5	ENDO/GP	124	0.29	4.1	0.13
	Febx	2	0.14	40.1	0.64
Hole 6	ENDO/GP	154	0.16	14.4	0.20
	Febx	97	0.32	15.6	0.54
Hole 7	ENDO	28	0.02	54.2	0.96
Hole 7B	ENDO/GP	67	0.45	30.7	1.22
	Febx	113	0.07	15.1	0.83
Hole 8	ENDO/GP	57	0.03	16.7	0.32
Hole 9	ENDO/GP	56	0.31	24.0	0.50
	Febx	2	0.01	24.4	0.30

	Lithology	Mass (kg)	Au (g/mt)	Ag (g/mt)	Cu (%)
	ENDO/EXO/QFP/GP	1013	0.16	17.8	0.48
	Mag	59	0.13	22.1	1.75
	Febx	281	0.15	13.7	0.56

From: Ryan Allen
Sent: Thursday, March 23, 2023 10:26 AM
To: Deepak Malhotra <Deepak@RDIminerals.com>
Subject: Empire Composites

Hello Deepak,
I wanted to check in on your thoughts regarding the compositing/sample prep for the Empire project. The first delivery of 7 holes is similar to what you had requested, but the second delivery had more intervals in holes 7B and 8 that will dilute the expected copper grade. I have included a table that describes the requested/expected values you had requested based on equal weights versus the actual weighed average of received intervals.

Would you like to use the extra samples in holes 7B and 8 or omit them? Also, what percentages do you want to use from each hole to make the master composites? Can we verify which holes are Mag and which holes are Endo/Exo?

Thanks,
Ryan

	Intervals		Total Copper Grade (%)	
	Expected	Actual	Expected	Actual
Hole 1	43	40	0.92	0.61
Hole 2	36	34	0.07	0.07
Hole 3	36	36	0.90	0.95
Hole 4	23	23	0.73	0.52
Hole 5	34	34	0.21	0.14
Hole 6	46	46	0.41	0.33
Hole 7	8	8	0.73	0.96
Hole 7B	36	52	1.43	0.98
Hole 8	8	18	0.60	0.32
Hole 9	12	14	0.55	0.50

Re: Core List Required for Met Tests?

David Jarvis <david.jarvis@phoenixcopper.com>
Tue 2/14/2023 11:40 AM
To:Deepak Malhotra <dmalhotra@fortedynamics.com>

Deepak,

I have reviewed your suggested intervals with Ryan and Keian, we are all in agreement.

Much of this core is highly fractured, producing a representative ¼ cut will be difficult. As such, Konnex will be bagging the "half core". This should provide you with extra material should it be required.

We expect the core to arrive in Golden late on the afternoon of the 22nd. For the security of the core, we would like to park our vehicle inside Forte's fenced area for the night.

I'm assuming that the core will be offloaded and inventoried on the morning of the 23rd.

For the meeting on the 23rd we would like to start at noon as Ahmet will be traveling from SLC that morning. If possible, we would like to have some sort of sandwiches in the office for lunch to insure the best use of our time together.

Please feel free to call or email and questions or concerns.

Regards,
Dave

From: Deepak Malhotra <dmalhotra@fortedynamics.com>
Sent: Tuesday, February 14, 2023 8:29 AM
To: David Jarvis <david.jarvis@phoenixcopper.com>
Cc: Ryan Allen <ryan@rdiminerals.com>; Nick Gow <ngow@fortedynamics.com>; Barry Carlson <bcarlson@fortedynamics.com>
Subject: Re: Core List Required for Met Tests?

Dave
I have spent time reviewing / analyzing the drill core data for the seven holes. Assuming equal weights of assayed intervals , the average assay of each hole was as follows:

1. 22-01: 0.917%Cu (43 intervals)
2. 22-02:0.071%Cu (36 intervals)

3. 22-03:0.895%Cu (36 intervals)
4. 22-04:0.73%Cu (0 ft to 114 ft; neglect 114 ft to 364.5 ft) (23 intervals)
5. 22-05: 0.207%Cu (34 intervals)
6. 22-06:0.406%Cu (0 ft to 250 ft; neglect 250 ft to 817 ft) (46 intervals)
7. 22-07:0.725%Cu (8 intervals)

The average copper grade of equally weighted seven holes is 0.56%Cu. This will represent the master composite. We can use composites from holes 1 and 3 for high grade and composites for 5 and 6 for low grade in geo-met testing.
If you have any questions, please do not hesitate to call me.
Looking forward to seeing you on 22nd.
Regards
Deepak Malhotra
Director of Metallurgy
Forte Dynamics
Cell:720-261-2450

From: David Jarvis <david.jarvis@phoenixcopper.com>
Sent: Monday, February 13, 2023 4:51 PM
To: Deepak Malhotra <dmalhotra@fortedynamics.com>
Subject: Core List Required for Met Tests?

Hello Deepak,

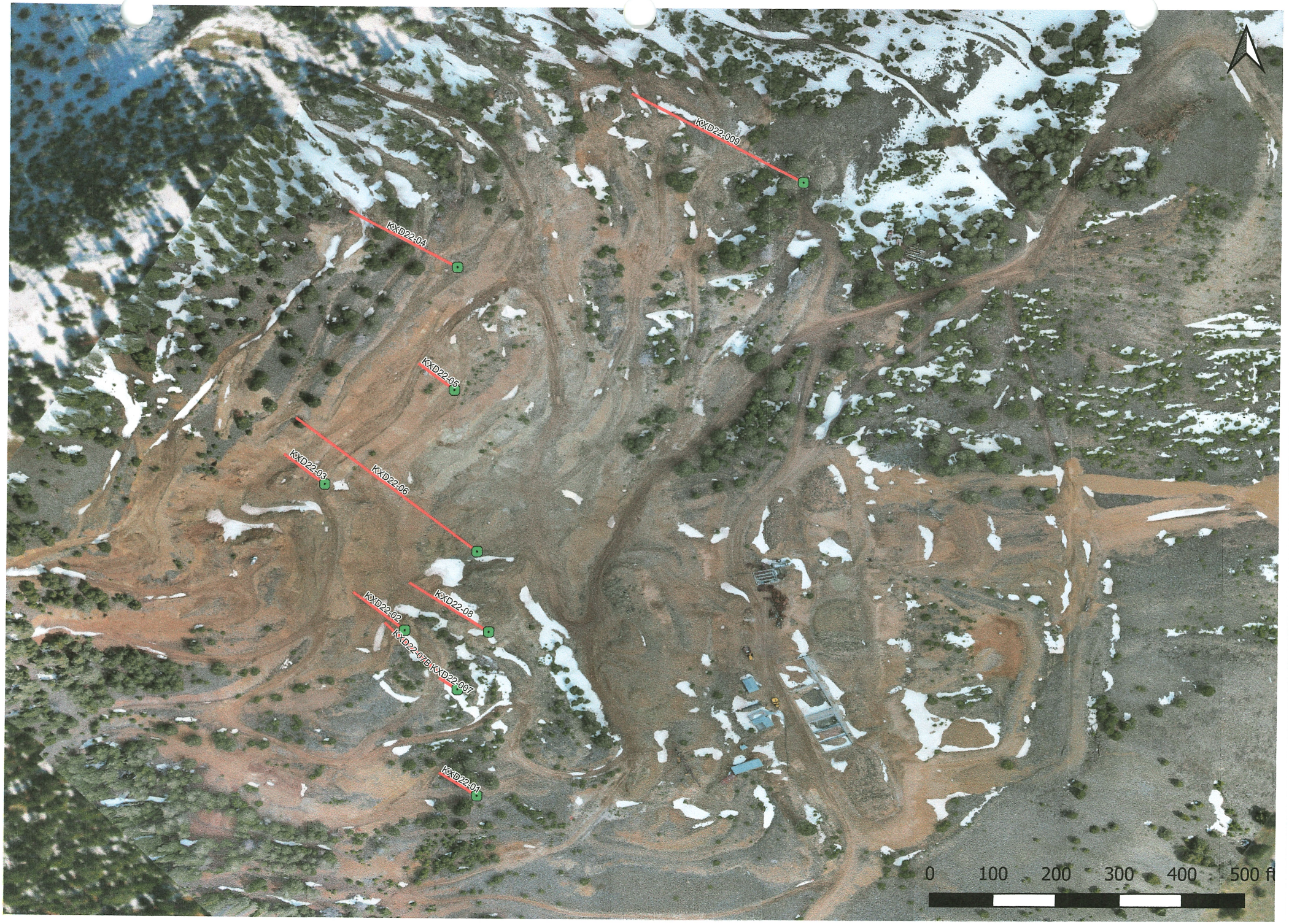
I hope this finds you well.

By chance is there any progress on choosing core intervals for the Empire met tests?

Unfortunately, Konnex is still waiting on the results for the last holes from ALS Labs. Hopefully your people will find sufficient core intervals within the hole data supplied last week to perform the tests. This week we have an extra geologist on hand to pull the core once specified.

To follow up on your crushing index question, I was surprised when Ryan told me that a crushing index test has not been performed for Empire rock. Once the snow melts in the AP Pit, within 60days, we will collect the samples and get them to you.

Regards,
Dave


KXD22-009
KXD22-04
KXD22-05
KXD22-03
KXD22-06
KXD22-08
KXD22-02
KXD22-07B
KXD22-007
KXD22-01
0
100
200
300
400
500 ft

23009 Empire Composites

Hole #1 Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200001MET	3.2907	0.748
2200002MET	4.941	1.162
2200003MET	3.6922	0.911
2200004MET	4.1617	1.001
2200005MET	0.9793	0.903
2200007MET	7.4029	1.731
2200008MET	3.3743	0.872
2200009MET	2.8669	0.715
2200010MET	2.5359	0.731
2200011MET	4.2354	0.983
2200013MET	4.4899	1.139
2200014MET	6.9801	1.723
2200020MET	3.355	0.862
2200022MET	1.3363	0.404
2200026MET	3.3265	0.799
2200028MET	9.6942	2.475
2200029MET	10.5035	2.822
2200030MET	8.9625	2.589
2200032MET	10.6233	2.596
2200033MET	7.7882	1.924
2200034MET	1.6718	0.408
2200036MET	8.0703	1.950
2200038MET	10.6453	2.424
2200039MET	2.4939	0.551
2200040MET	5.046	1.226
2200042MET	3.7656	0.970
2200044MET	3.6421	0.874
2200045MET	0.9757	0.172
2200046MET	1.6992	0.366
Total	142.5	36.029
Quarter Split	35.6	

Hole #1 Mag Composite

Sample	Start Wt (kg)	1/2 split 6 mesh Wt (kg)	
2200016MET	1.555	0.705	Mag Skarn
2200019MET	0.8238	0.409	Mag Skarn
2200021MET	3.9579	1.751	Mag Skarn
2200035MET	0.8376	0.385	Mag Skarn
2200041MET	2.863	1.315	Mag Skarn
2200023MET	1.658	0.694	Iron Breccia or Granite Mag
2200027MET	1.6524	0.776	Iron Breccia or Granite Mag
Total	13.3	6.035	
Half Split	6.7		

Retain

2200047MET	6.0193
2200048MET	7.6637
2200049MET	10.3256
2200051MET	1.5226

23009 Empire Composites

Hole #2 Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200052MET	4.6642	1.158
2200053MET	1.9996	0.414
2200054MET	5.631	1.213
2200055MET	2.232	0.494
2200057MET	1.7877	0.410
2200065MET	5.5999	1.493
2200066MET	2.811	0.666
2200067MET	2.0567	0.489
2200068MET	1.707	0.383
2200069MET	8.6252	2.164
2200072MET	8.4012	2.272
2200073MET	10.328	2.450
2200074MET	11.7749	3.155
2200075MET	10.7408	2.661
2200076MET	9.6473	2.500
2200078MET	4.0366	1.068
2200079MET	7.3755	1.796
2200080MET	6.0663	1.439
2200082MET	3.7085	1.239
2200083MET	3.2866	0.749
2200085MET	4.6812	1.021
2200087MET	5.8947	1.574
2200088MET	5.343	1.426
2200089MET	12.8118	3.468
2200090MET	8.4035	2.105
Total	149.6	37.805
Quarter Split	37.4	

Hole #2 Mag Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)	
2200059MET	4.2166	1.084	Mag Skarn
2200081MET	9.4303	2.406	Mag Skarn
2200086MET	7.2527	1.965	Mag Skarn
2200058MET	3.6694	0.894	Iron Breccia or Granite Mag
2200060MET	6.6557	1.823	Iron Breccia or Granite Mag
2200061MET	6.8093	1.758	Iron Breccia or Granite Mag
2200062MET	7.4714	1.833	Iron Breccia or Granite Mag
2200063MET	5.8531	1.603	Iron Breccia or Granite Mag
2200070MET	3.6833	0.912	Iron Breccia or Granite Mag
Total	55	14.278	
Quarter Split	13.8		

23009 Empire Composites

Hole #3 Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200101MET	6.2187	1.626
2200102MET	6.291	1.602
2200103MET	2.8254	0.682
2200104MET	3.82	0.876
2200106MET	1.3666	0.287
2200107MET	1.2079	0.250
2200109MET	4.4772	1.049
2200110MET	0.7333	0.176
2200112MET	1.0808	0.236
2200120MET	3.8858	1.059
2200121MET	3.2319	0.782
2200122MET	6.0252	1.519
2200123MET	3.5485	0.863
2200124MET	4.3989	1.150
2200125MET	5.8463	1.383
2200127MET	3.7366	1.019
2200128MET	2.6119	0.697
2200132MET	1.2971	0.302
Total	62.6	15.559
Quarter Split	15.7	

Hole #3 Mag Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)	
2200091MET	2.024	0.482	Mag Skarn
2200093MET	6.3348	1.694	Mag Skarn
2200094MET	5.3096	1.482	Mag Skarn
2200095MET	3.5401	0.947	Mag Skarn
2200096MET	2.7147	0.629	Mag Skarn
2200100MET	7.7675	1.728	Mag Skarn
2200108MET	0.7949	0.161	Mag Skarn
2200097MET	5.8972	1.496	Iron Breccia or Granite Mag
2200098MET	5.4719	1.376	Iron Breccia or Granite Mag
2200113MET	1.5711	0.378	Iron Breccia or Granite Mag
2200114MET	4.0879	1.000	Iron Breccia or Granite Mag
2200115MET	0.7855	0.167	Iron Breccia or Granite Mag
2200116MET	1.3583	0.321	Iron Breccia or Granite Mag
2200117MET	3.2459	0.794	Iron Breccia or Granite Mag
2200119MET	2.5106	0.632	Iron Breccia or Granite Mag
2200129MET	5.1862	1.274	Iron Breccia or Granite Mag
2200130MET	3.5152	0.806	Iron Breccia or Granite Mag
2200131MET	3.3517	0.843	Iron Breccia or Granite Mag
Total	65.467	16.211	
Quarter Split	16.367		

23009 Empire Composites

Hole #4 Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200141MET	2.2473	0.570
2200142MET	8.6827	2.015
2200143MET	4.0916	1.033
2200144MET	2.2741	0.590
2200145MET	6.6777	1.628
2200147MET	1.9193	0.477
2200148MET	1.2612	0.307
2200150MET	3.9873	0.923
2200151MET	4.0963	1.059
2200152MET	9.6262	2.188
2200154MET	9.5674	2.294
2200155MET	11.1873	2.964
2200156MET	10.0245	2.786
2200157MET	8.0767	1.828
2200158MET	8.1471	2.181
2200159MET	8.5747	2.162
2200161MET	8.0093	2.041
2200162MET	11.3123	2.851
2200163MET	9.7948	2.205
2200164MET	14.5753	3.942
2200165MET	4.1584	1.019
2200166MET	10.2599	2.451
2200167MET	12.2188	2.888
Total	170.8	42.401
Quarter Split	42.7	

23009 Empire Composites

Hole #5 Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200232MET	4.645	1.339
2200233MET	4.543	1.221
2200234MET	1.7642	0.448
2200235MET	3.5073	0.887
2200236MET	5.9906	1.867
2200237MET	3.5824	0.952
2200239MET	2.6171	0.691
2200240MET	2.0819	0.547
2200241MET	4.0622	1.087
2200242MET	5.9205	1.587
2200243MET	3.2556	0.802
2200244MET	2.5716	0.612
2200246MET	1.8249	0.436
2200248MET	3.7873	0.983
2200249MET	2.9096	0.794
2200250MET	3.7828	0.994
2200252MET	3.0226	0.688
2200253MET	2.9069	0.732
2200254MET	3.297	0.837
2200255MET	1.0876	0.245
2200256MET	1.6102	0.356
2200257MET	0.9871	0.234
2200258MET	2.1921	0.462
2200259MET	6.6505	1.755
2200261MET	5.61	1.495
2200262MET	5.8056	1.606
2200263MET	4.4491	1.186
2200264MET	2.5294	0.577
2200265MET	4.7515	1.081
2200266MET	6.5355	1.648
2200268MET	5.6737	1.556
2200269MET	4.8035	1.136
2200270MET	5.5855	1.628
Total	124.3	32.465
Quarter Split	31.1	

Retain

2200247MET	2.0825

23009 Empire Composites

Hole #6 Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200271MET	1.9059	0.386
2200272MET	1.0932	0.181
2200273MET	1.0884	0.248
2200283MET	2.8625	0.608
2200291MET	5.9522	1.166
2200292MET	2.4631	0.613
2200293MET	5.4787	1.341
2200294MET	2.7554	0.683
2200295MET	3.0677	0.782
2200297MET	6.9784	1.780
2200298MET	4.1126	1.096
2200299MET	7.059	1.993
2200300MET	2.6048	0.583
2200301MET	6.4269	1.753
2200302MET	5.1886	1.340
2200304MET	5.2883	1.331
2200305MET	5.7509	1.471
2200306MET	4.7796	1.202
2200307MET	6.3092	1.433
2200308MET	7.0441	2.145
2200309MET	6.3148	1.452
2200311MET	4.8551	1.219
2200312MET	7.1435	1.934
2200313MET	5.1979	1.382
2200314MET	6.3742	1.857
2200315MET	6.8229	1.854
2200316MET	6.4772	1.796
2200318MET	7.1335	2.048
2200319MET	8.7457	2.168
2200323MET	6.9853	1.761
Total	154.3	39.604
Quarter Split	38.6	

Hole #6 Mag Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)	
2200274MET	1.2386	0.281	Iron Breccia or Granite Mag
2200276MET	2.0139	0.505	Iron Breccia or Granite Mag
2200277MET	2.2484	0.558	Iron Breccia or Granite Mag
2200278MET	2.3727	0.522	Iron Breccia or Granite Mag
2200279MET	7.5003	2.047	Iron Breccia or Granite Mag
2200280MET	8.5106	2.139	Iron Breccia or Granite Mag
2200281MET	6.391	1.620	Iron Breccia or Granite Mag
2200284MET	11.0153	2.845	Iron Breccia or Granite Mag
2200285MET	10.5264	2.916	Iron Breccia or Granite Mag
2200286MET	6.9194	1.812	Iron Breccia or Granite Mag
2200287MET	7.6542	1.977	Iron Breccia or Granite Mag
2200288MET	3.8182	0.933	Iron Breccia or Granite Mag
2200290MET	6.5933	1.664	Iron Breccia or Granite Mag
2200320MET	7.1461	1.638	Iron Breccia or Granite Mag
2200321MET	5.401	1.251	Iron Breccia or Granite Mag
2200322MET	7.8658	2.031	Iron Breccia or Granite Mag
Total	97.215	24.737	
Quarter Split	24.304		

23009 Empire Composites

Hole #7 Exo/Endo Composite

Sample	Start Wt (kg)	1/2 split 6 mesh Wt (kg)
2200456MET	3.187	1.531
2200457MET	2.8602	1.451
2200458MET	1.1077	0.390
2200459MET	4.4006	2.168
2200461MET	4.7466	2.575
2200462MET	3.3688	1.634
2200464MET	3.8872	1.919
2200465MET	4.8723	2.516
Total	28.4	14.184
Half Split	14.2	

23009 Empire Composites

Hole #7B Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200469MET	0.9697	0.195
2200470MET	1.5458	0.339
2200471MET	3.9937	0.752
2200473MET	3.8994	0.724
2200474MET	1.2957	0.302
2200475MET	3.4612	0.779
2200476MET	3.5779	0.766
2200477MET	5.05	1.377
2200478MET	3.3594	0.962
2200479MET	2.0741	0.452
2200480MET	2.3782	0.643
2200482MET	2.7545	0.694
2200483MET	3.8888	1.017
2200529MET	5.6029	1.538
2200531MET	7.0925	1.964
2200532MET	4.6462	1.155
2200533MET	5.8132	1.439
2200546MET	5.4656	1.341
Total	66.9	16.437
Quarter Split	16.7	

Hole #7B Mag Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)	
2200484MET	3.504	0.811	Iron Breccia or Granite Mag
2200485MET	0.993	0.255	Iron Breccia or Granite Mag
2200486MET	1.3555	0.309	Iron Breccia or Granite Mag
2200487MET	1.5518	0.389	Iron Breccia or Granite Mag
2200489MET	3.8518	1.021	Iron Breccia or Granite Mag
2200490MET	4.3989	1.141	Iron Breccia or Granite Mag
2200491MET	4.7785	1.244	Iron Breccia or Granite Mag
2200492MET	2.1007	0.514	Iron Breccia or Granite Mag
2200493MET	0.6906	0.152	Iron Breccia or Granite Mag
2200494MET	3.4127	0.903	Iron Breccia or Granite Mag
2200496MET	3.3166	0.721	Iron Breccia or Granite Mag
2200497MET	6.1146	1.569	Iron Breccia or Granite Mag
2200498MET	3.4199	0.822	Iron Breccia or Granite Mag
2200499MET	1.5019	0.363	Iron Breccia or Granite Mag
2200500MET	3.234	0.715	Iron Breccia or Granite Mag
2200501MET	2.8134	0.738	Iron Breccia or Granite Mag
2200503MET	2.4835	0.569	Iron Breccia or Granite Mag
2200504MET	2.9072	0.728	Iron Breccia or Granite Mag
2200505MET	1.8351	0.443	Iron Breccia or Granite Mag
2200506MET	1.0399	0.184	Iron Breccia or Granite Mag
2200507MET	1.6586	0.401	Iron Breccia or Granite Mag
2200508MET	1.2487	0.269	Iron Breccia or Granite Mag
2200510MET	1.5975	0.415	Iron Breccia or Granite Mag
2200511MET	4.9215	1.312	Iron Breccia or Granite Mag
2200513MET	6.3407	1.626	Iron Breccia or Granite Mag
2200514MET	4.9299	1.144	Iron Breccia or Granite Mag
2200515MET	5.2147	1.355	Iron Breccia or Granite Mag
2200517MET	3.4717	0.819	Iron Breccia or Granite Mag
2200518MET	2.76	0.650	Iron Breccia or Granite Mag
2200519MET	2.2793	0.414	Iron Breccia or Granite Mag
2200520MET	7.8315	1.848	Iron Breccia or Granite Mag
2200521MET	4.3601	1.030	Iron Breccia or Granite Mag
2200522MET	4.2586	1.069	Iron Breccia or Granite Mag
2200525MET	7.0393	1.781	Iron Breccia or Granite Mag
Total	113.216	27.725	
Quarter Split	28.304		

23009 Empire Composites

Hole #8 Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200572MET	1.751	0.407
2200573MET	1.3635	0.333
2200575MET	1.8165	0.405
2200576MET	4.4485	1.114
2200577MET	2.1895	1.667
2200578MET	3.8166	1.004
2200579MET	1.0967	0.240
2200585MET	5.2143	1.295
2200586MET	5.1877	1.306
2200587MET	5.7747	1.466
2200589MET	4.9727	1.234
2200590MET	3.4667	0.784
2200591MET	3.0031	0.752
2200592MET	0.9137	0.216
2200593MET	3.4883	0.973
2200594MET	3.4243	0.801
2200596MET	4.913	1.163
Total	56.8	15.160
Quarter Split	14.2	

23009 Empire Composites

Hole #9 Exo/Endo Composite

Sample	Start Wt (kg)	1/4 split 6 mesh Wt (kg)
2200658MET	1.7405	0.371
2200667MET	4.1828	0.926
2200673MET	3.2406	0.847
2200675MET	6.6687	1.464
2200676MET	3.1544	0.781
2200677MET	5.1859	1.154
2200678MET	5.4133	1.320
2200679MET	5.7889	1.253
2200682MET	2.4909	1.202
2200683MET	3.316	0.772
2200684MET	4.7512	0.969
2200685MET	2.7535	0.598
2200686MET	2.6876	0.719
2200691MET	6.3937	1.554
Total	56.0	13.930
Quarter Split	14.0	

APPENDIX B: HEAD ANALYSES OF COMPOSITE SAMPLES

Analytical Report

Batch ID: Head A-B
Report Date: 5/8/2023
Report to: Ryan Allen

Project 23009

Lab ID	Sample Name	Gold mg/kg	Silver mg/kg
FAE23-1147	23009 Endo/Exo Head A	0.16	22.7
FAE23-1148	23009 Endo/Exo Head B	0.12	14.7
FAE23-1149	23009 Mag Comp Head A	0.14	13.7
FAE23-1150	23009 Mag Comp Head B	0.15	15.0
	MB	<0.02	<0.3
CRM	.056 mg/kg Au	0.049	12.2

Analysis Method: litharge fire assay fusion with aqua regia dissolution of prill and final analysis by AAS for gold. Silver by calculation

Reporting Limit:	1AT	1/2 AT
Gold (mg/kg):	0.02	0.04
Silver (mg/kg):	0.3	0.6

Reviewed and Approved by: _____

Jessica Axen

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc
11475 West I-70 Frontage Road North
Wheat Ridge, CO 80033
Attention: Jessica Axen

Laboratory No.: 232032
Client Number: F174
Date Received: 10 Nov 2023
Date Completed: 20 Nov 2023
PO # 23009

Method: 4-Acid Digestion, AAS analysis.

Lab code:	7015	7015
Element:	Copper	Copper
Detection Limit:	0.01	0.01
Units:	%	%
23009 Endo/Evo MC Head	0.43	0.42
23009 Mag MC Head	0.74	0.77
Blank	<0.01	
OREAS 504b	1.04	
Certified Reference Material	**Cu**	
OREAS 504b	1.11	
95% Confidence Interval of Mean	0.04	

Karen Boldi

Karen Boldi, Laboratory Technician

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted by: Resource Development, Inc.
11475 West I-70 Frontage Road North
Wheat Ridge, Colorado 80033
Attention: Jessica Axen

Laboratory No.: 232032
Client Number: F174
Date Received: 10 Nov 2023
Date Completed: 01 Dec 2023

Method: 4 Acid digestion, ICP analysis, Ag by AA analysis.
PO # 23009

Lab code: 7045

Element:	Ag	Al	As	Ba	Bi	Ca	Cd	Co	Cr	Cu	Fe	K	Mg	Mn	Mo	Na	Ni	Pb	Sr	Ti	V	W	Zn
Detection Limit:	0.2	0.01	10	1	10	0.01	1	1	1	0.01	0.01	0.01	0.01	0.01	1	0.01	5	10	5	0.01	1	10	0.01
Reporting Unit:	mg/kg	%	mg/kg	mg/kg	mg/kg	%	mg/kg	mg/kg	mg/kg	%	%	%	%	%	mg/kg	%	mg/kg	mg/kg	mg/kg	%	mg/kg	mg/kg	%
23009 Endo/Evo MC Head	19.4	5.21	112	606	<10	11.62	23	23	96	0.43	8.93	2.63	1.03	0.27	62	0.68	25	138	261	0.22	62	174	0.32
23009 Mag MC Head	19.4	3.23	316	286	90	5.18	51	41	98	0.74	24.55	0.52	0.35	0.47	177	0.05	45	177	87	0.17	77	193	0.46

Karen Boldi

Karen Boldi, Laboratory Technician

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted by: Resource Development, Inc.
11475 West I-70 Frontage Road North
Wheat Ridge, Colorado 80033
Attention: Jessica Axen

Laboratory No.: 232032
Client Number: F174
Date Received: 10 Nov 2023
Date Completed: 01 Dec 2023

Method: Lithium metaborate fusion, ICP analysis.
PO # 23009

Lab Code: 7046

Element:	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	Na_2O	K_2O	TiO_2	MnO	SrO	BaO	Cr_2O_3	P_2O_5	$LOI_{1000\,^{o}C}$	SUM
Detection Limit:	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.10	
Sample ID:	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%
23009 Endo/Evo MC H	53.31	10.04	12.13	14.29	1.62	0.91	2.62	0.37	0.38	0.02	0.07	0.01	0.30	4.23	100.29
23009 Mag MC Head	43.95	5.92	34.27	6.50	0.59	0.07	0.59	0.29	0.54	<0.01	0.03	0.02	0.55	5.94	99.26

Karen Boldi

Karen Boldi, Laboratory Technician

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

APPENDIX C: COMMINUTION TEST RESULTS

Engineer		**Test ID**	**Mag**
Technician	Robert Davis	**Description**	Abrasion Index
Project Name	Empire/Konnex	**Date**	7/14/2023
Project No.	23009		

400	
Split 1 Weight, gram	402.13
Split 2 Weight, gram	402.77
Split 3 Weight, gram	400.55
Split 4 Weight, gram	402.32

Testing Equipment	
Rotor RPM	631
Specification	632
Drum RPM	72
Specification	70

Bond Abrasion Index		
Starting Paddle Weight, gram	94.3882	(4 decimals)
Final Washed Paddle Weight, gram	94.2316	
Ai	0.1566	

FORTE ANALYTICAL	**Engineer**		**Test ID**	**Endo/Exo**
	Technician	Robert Davis	**Description**	Abrasion Index
	Project Name	Empire/Konnex	**Date**	7/13/2023
	Project No.	23009		

400	
Split 1 Weight, gram	400.49
Split 2 Weight, gram	401.54
Split 3 Weight, gram	400.78
Split 4 Weight, gram	402.50

Testing Equipment	
Rotor RPM	631
Specification	632
Drum RPM	72
Specification	70

Bond Abrasion Index		
Starting Paddle Weight, gram	94.4655	(4 decimals)
Final Washed Paddle Weight, gram	94.3882	
Ai	0.0773	

Engineer		Test ID	**Mag**	
Technician	Robert Davis	Date	7/14/2023	
Project Name	Empire/ Konnex	# of Qualifying Specimens	11	H20 Fill Line=900mL
Project No.	23009			G=A/900-(B-A)

Rock No.	Specimen Width	Impact Energy	Energy per Width	Specimen Width	Impact Energy	Energy per Width
	Inch	Ft-lb	Ft-lb/in	mm	Joule	Joule/mm
1	1.69	40	23.70	42.86	54	1.27
2	1.88	30	16.00	47.63	41	0.85
3	1.63	55	33.85	41.28	75	1.81
4	2.00	35	17.50	50.80	47	0.93
5	1.50	35	23.33	38.10	47	1.25
6	2.00	40	20.00	50.80	54	1.07
7	1.50	65	43.33	38.10	88	2.31
8	1.63	25	15.38	41.28	34	0.82
9	1.50	25	16.67	38.10	34	0.89
10	1.56	30	19.20	39.69	41	1.02
11	1.63	25	15.38	41.28	34	0.82
12						
13						
14						
15						
16						
17						
18						
19						
20						

A	B	G
Wt. of Rock	Wt. of H20+Sample	Specific Gravity
Grams	Grams	
545	1317.5	4.27
405	1203	3.97
488.5	1269.5	4.11
651.5	1379.5	3.79
310	1135.5	4.16
696.5	1409.5	3.72
422	1220	4.14
713	1423.5	3.76
246	1055.5	2.72
408	1188	3.4
249.5	1047.5	2.45

Bond Crusher Work Index			
Average Impact Energy per Width	22.21	Average Impact Energy per Width	1.19
Specific Gravity			
Specific Gravity	3.68		
CWi, kWh/st	15.63	CWi, kWh/mt	17.23

Engineer		Test ID	Endo/Exo
Technician	Robert Davis	Date	7/12/2023
Project Name	Empire/ Konnex	# of Qualifying Specimens	11
Project No.	23009		

H2O Fill Line=900mL

G=A/900-(B-A)

Rock No.	Specimen Width	Impact Energy	Energy per Width	Specimen Width	Impact Energy	Energy per Width
	Inch	Ft-lb	Ft-lb/in	mm	Joule	Joule/mm
1	1.88	30	16.00	47.63	41	0.85
2	1.75	50	28.57	44.45	68	1.53
3	1.50	45	30.00	38.10	61	1.60
4	1.63	30	18.46	41.28	41	0.99
5	1.25	20	16.00	31.75	27	0.85
6	1.25	20	16.00	31.75	27	0.85
7	1.50	25	16.67	38.10	34	0.89
8	2.00	20	10.00	50.80	27	0.53
9	1.75	15	8.57	44.45	20	0.46
10	2.00	40	20.00	50.80	54	1.07
11	1.56	15	9.60	39.69	20	0.51
12						
13						
14						
15						
16						
17						
18						
19						
20						

A	B	G
Wt.of Rock	Wt. of H2O+Sample	Specific Gravity
Grams	Grams	
382	1156.5	3.04
627	1338.5	3.33
367.5	1155.5	3.28
384.5	1161.5	3.13
201.5	1028.5	2.76
344	1130.5	3.03
279.5	1093	3.23
466.5	1227	3.34
241	1062	3.05
862.5	1502.5	3.32
293.5	1097	3.04

Bond Crusher Work Index			
Average Impact Energy per Width	17.26	Average Impact Energy per Width	0.92
Specific Gravity			
Specific Gravity	3.14		
CWi, kWh/st	14.23	CWi, kWh/mt	15.69

Bond Ball Mill

Client: **Empire Mines**
Sample ID: Endo/Exo Master Composite (100 mesh)

SIZE		MILL FEED			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	-	-	100.0
8	2,360	59.4	29.1	29.1	70.9
10	1,700	40.6	19.9	49.0	51.0
14	1,180	38.6	18.9	68.0	32.0
20	850	20.0	9.8	77.8	22.2
28	600	13.8	6.8	84.6	15.4
35	425	8.6	4.2	88.8	11.2
48	300	5.7	2.8	91.6	8.4
65	212	3.6	1.8	93.3	6.7
100	150	2.8	1.4	94.7	5.3
Pan		10.7	5.3	100.0	
Total		203.8	100.0		

Cycle #	grams / revolution
n-2	1.291
n-1	1.307
n	1.295
Average	1.298

Mill Feed Weight (grams)	1326
Desired Mesh of Grind:	100
Desired Micron of Grind:	150
Circulating Load (%)	250
Circulating Load (grams):	378.9

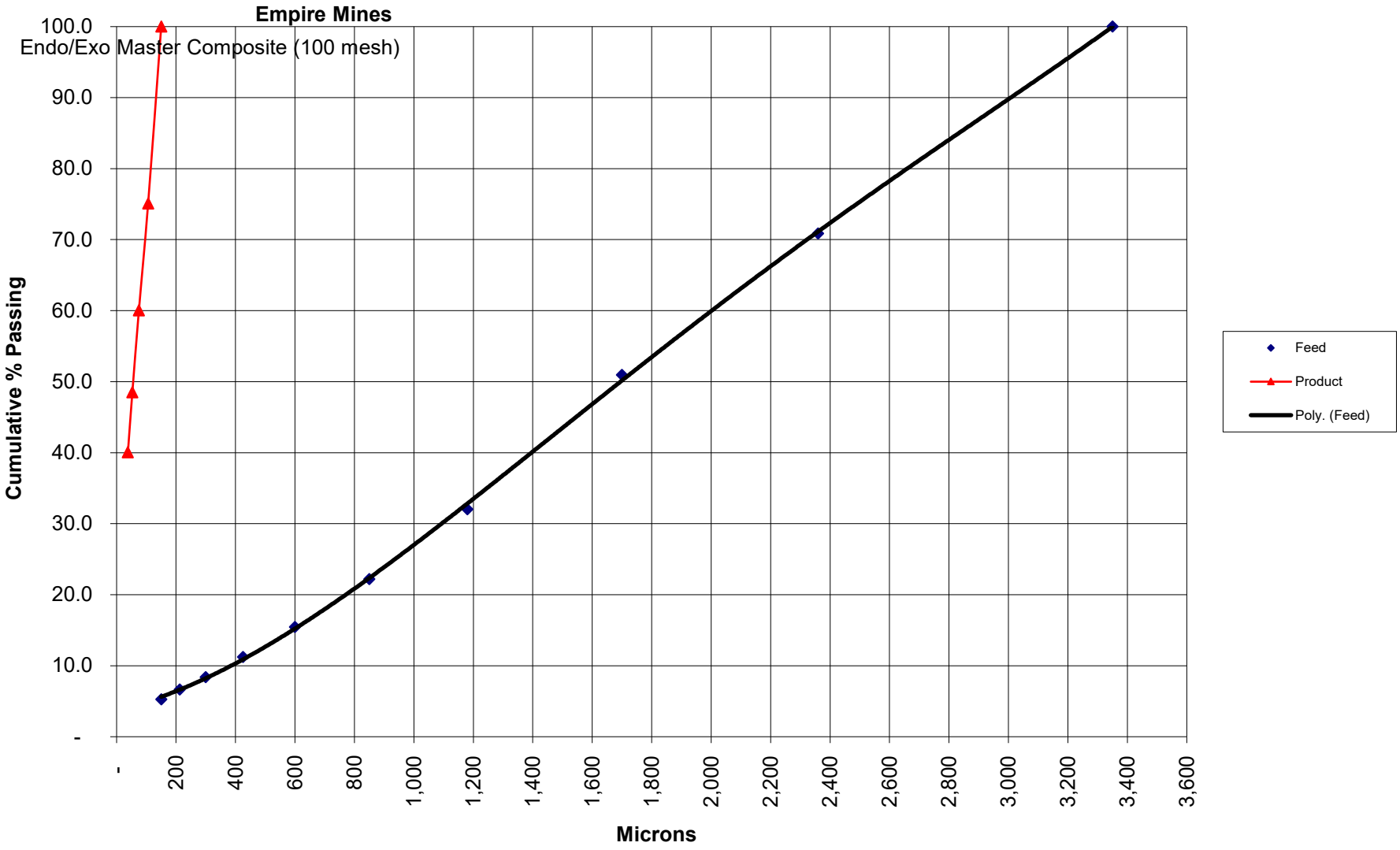
SIZE		GROUND PRODUCT			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
65	212	-	-	-	100.0
100	150	-	-	-	100.0
150	106	91.8	24.9	24.9	75.1
200	75	55.4	15.0	39.9	60.1
270	53	42.5	11.5	51.5	48.5
400	38	31.2	8.5	59.9	40.1
Pan	0	147.6	40.1	100.0	-
Total		368.5	100.0		

	Interpolated	Graphic
F80	2,670	2,641
P80	115	113

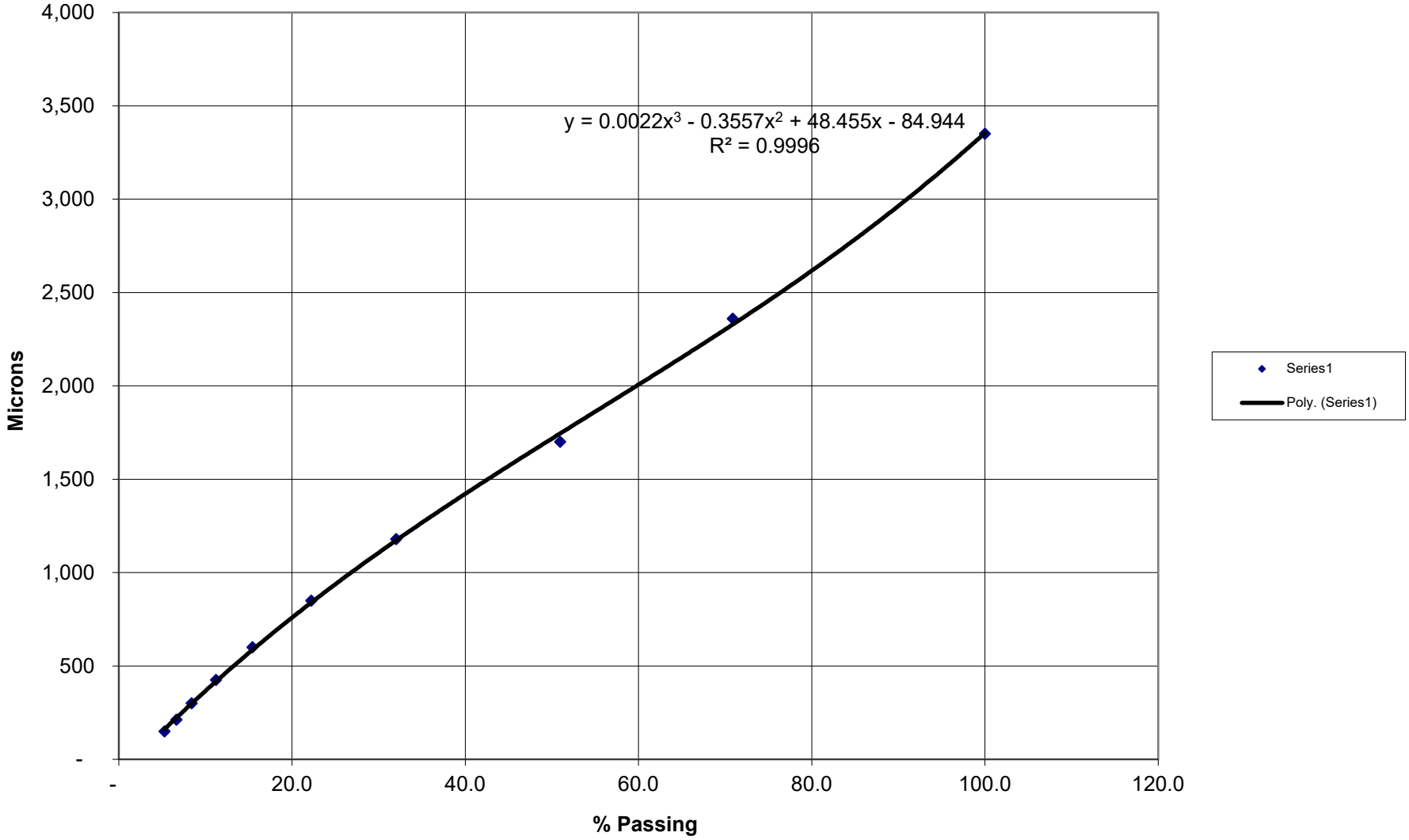
WORK INDEX - FROM GRAPH	15.24
WORK INDEX - INTERPOLATED	15.33

WORK INDEX - AVG	15.29

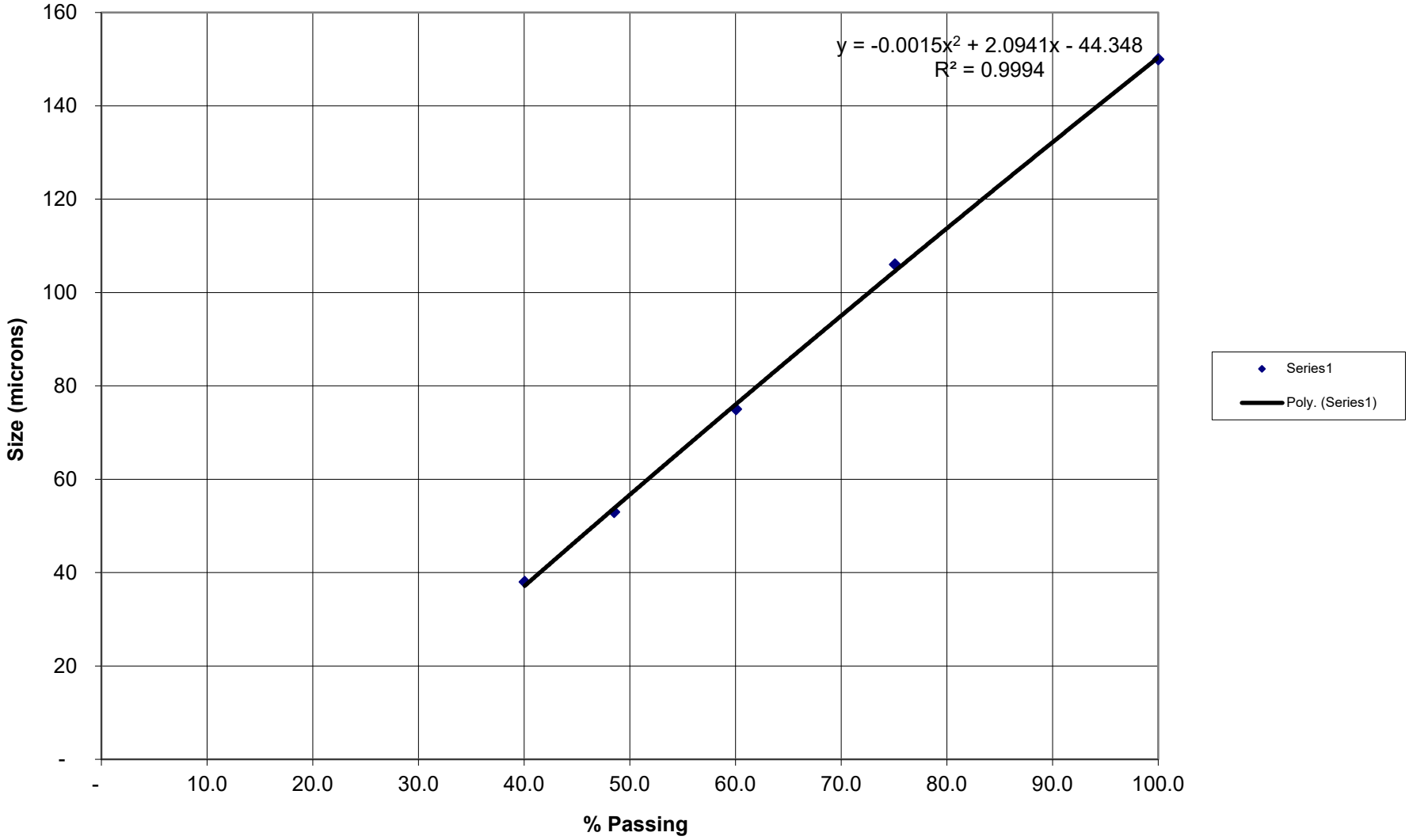
Size Analysis



Mill Feed Size Analysis



Mill Product Size Analysis



Bond Ball Mill

Feed $y = 0.0022x^3 - 0.3557x^2 + 48.455x - 84.944$

$R^2 = 0.9996$

Y	X
2641.376	80

	Microns	% Passing
6	3350	100.00
8	2360	70.87
10	1700	50.96
14	1180	32.02
20	850	22.21
28	600	15.44
35	425	11.22
48	300	8.41
100	150	5.27

Product $y = -0.0015x^2 + 2.0941x - 44.348$

$R^2 = 0.9994$

Y	X
113.332	80

Microns	% Passing
212	100.0
150	100.0
106	75.1
75	60.1
53	48.5
38	40.1

Average wt of 3 - 700 ml samples	
A1 wt (1)	1327.6
A1 wt (2)	1327.0
A1 wt (3)	1323.3
Average	1326.0

Y= A1 divided by 3.5 for 250% circulating load

X= average wt % of undersize from screening

Y=	378.8
X=	5.3

Variable	A	B	C	D	E	F	G
Calcs	average wt of 3 700 ml samples	A multiplied by X	Y minus B	C1 divided by 1.2	wt of product	E minus B	F divided by D
	A2 = E1, A3 = E2, etc.			C2 divided by G1			

		Undersize (g)				Mill Production (g)	
Cycle #	Feed (g)	In Feed	To be Produced	Mill Revs	Wt of Undersize (g)	Total Net	Per Rev
1	1326.0	69.8	309.0	263	444.0	374.2	1.423
2	444	23.4	355.5	250	350.8	327.4	1.310
3	350.8	18.5	360.4	275	394.9	376.4	1.369
4	394.9	20.8	358.0	262	357.5	336.7	1.287
5	357.5	18.8	360.0	280	380.0	361.2	1.291
6	380	20.0	358.8	278	383.1	363.1	1.307
7	383.1	20.2	358.7	274	375.7	355.5	1.295

SIZE		MILL FEED					
Tyler Mesh	Microns	Screen #1 Weight (g)	Screen #2 Weight (g)	Screen #3 Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	0.0	0.0	-	-	100.0
8	2,360	59.3	60.9	57.9	29.1	29.1	70.9
10	1,700	42.0	39.1	40.6	19.9	49.0	51.0
14	1,180	37.9	38.8	39.1	18.9	68.0	32.0
20	850	19.8	19.5	20.7	9.8	77.8	22.2
28	600	13.7	13.7	14.0	6.8	84.6	15.4
35	425	8.3	8.4	9.1	4.2	88.8	11.2
48	300	5.5	5.6	6.1	2.8	91.6	8.4
65	212	3.4	3.5	3.8	1.8	93.3	6.7
100	150	2.7	2.8	3.0	1.4	94.7	5.3
Pan		10.2	10.6	11.4	5.3	100.0	
Total		202.8	202.9	205.7	100.0		

Bond Ball Mill

Client: **Empire Mines**
Sample ID: Magnetic Composite (100 mesh)

SIZE		MILL FEED			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	-	-	100.0
8	2,360	58.4	27.5	27.5	72.5
10	1,700	39.5	18.6	46.1	53.9
14	1,180	35.9	16.9	63.1	36.9
20	850	19.5	9.2	72.2	27.8
28	600	14.1	6.7	78.9	21.1
35	425	10.3	4.8	83.7	16.3
48	300	7.7	3.6	87.4	12.6
65	212	5.4	2.5	89.9	10.1
100	150	4.5	2.1	92.0	8.0
Pan		16.9	8.0	100.0	
Total		212.2	100.0		

Cycle #	grams / revolution
n-2	1.428
n-1	1.484
n	1.465
Average	1.459

Mill Feed Weight (grams)	1200.9
Desired Mesh of Grind:	100
Desired Micron of Grind:	150
Circulating Load (%)	250
Circulating Load (grams):	343.1

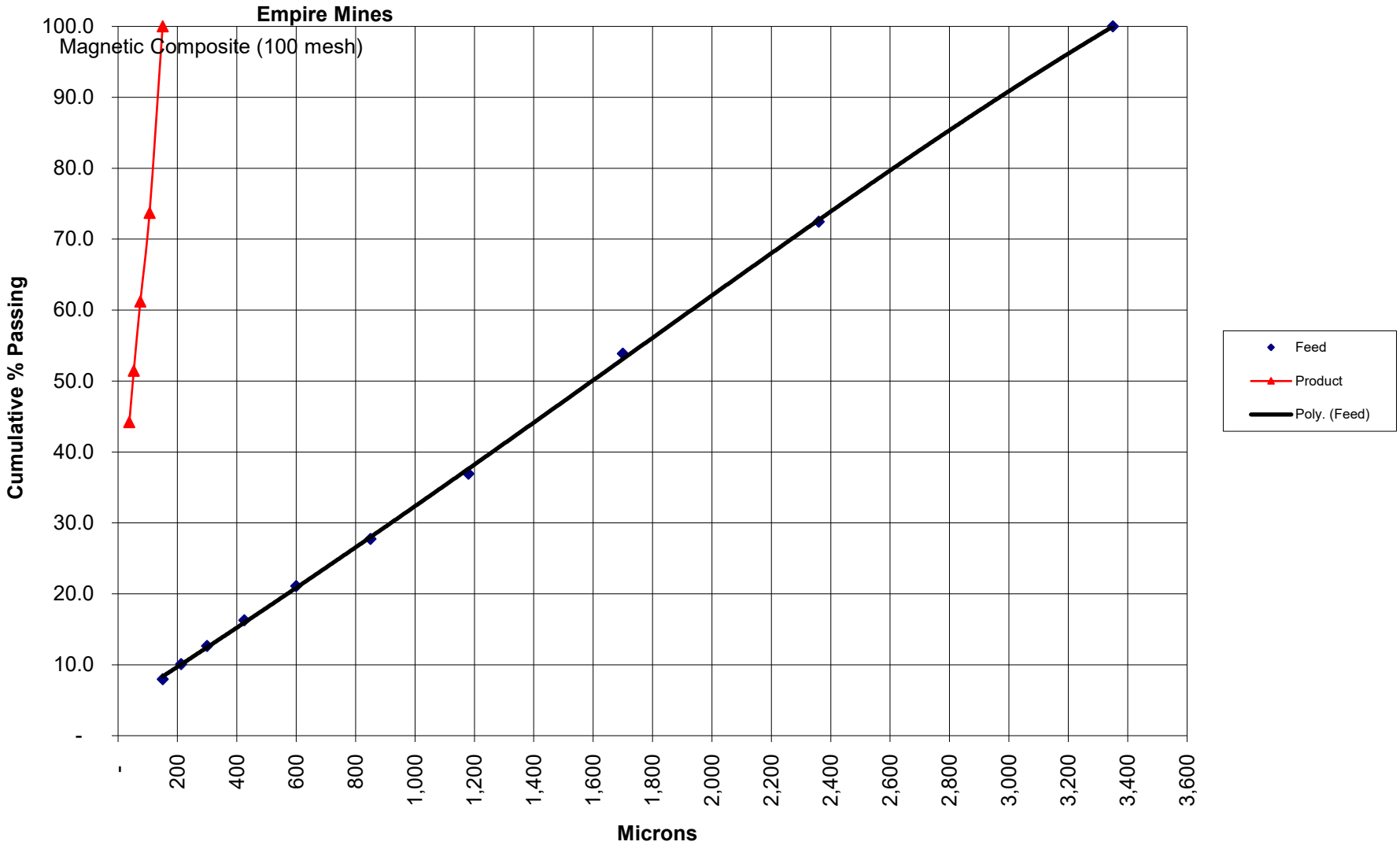
SIZE		GROUND PRODUCT			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
65	212	-	-	-	100.0
100	150	-	-	-	100.0
150	106	91.4	26.3	26.3	73.7
200	75	43.5	12.5	38.8	61.2
270	53	33.9	9.7	48.5	51.5
400	38	25.3	7.3	55.8	44.2
Pan	0	153.7	44.2	100.0	-
Total		347.8	100.0		

	Interpolated	Graphic
F80	2,631	2,614
P80	117	116

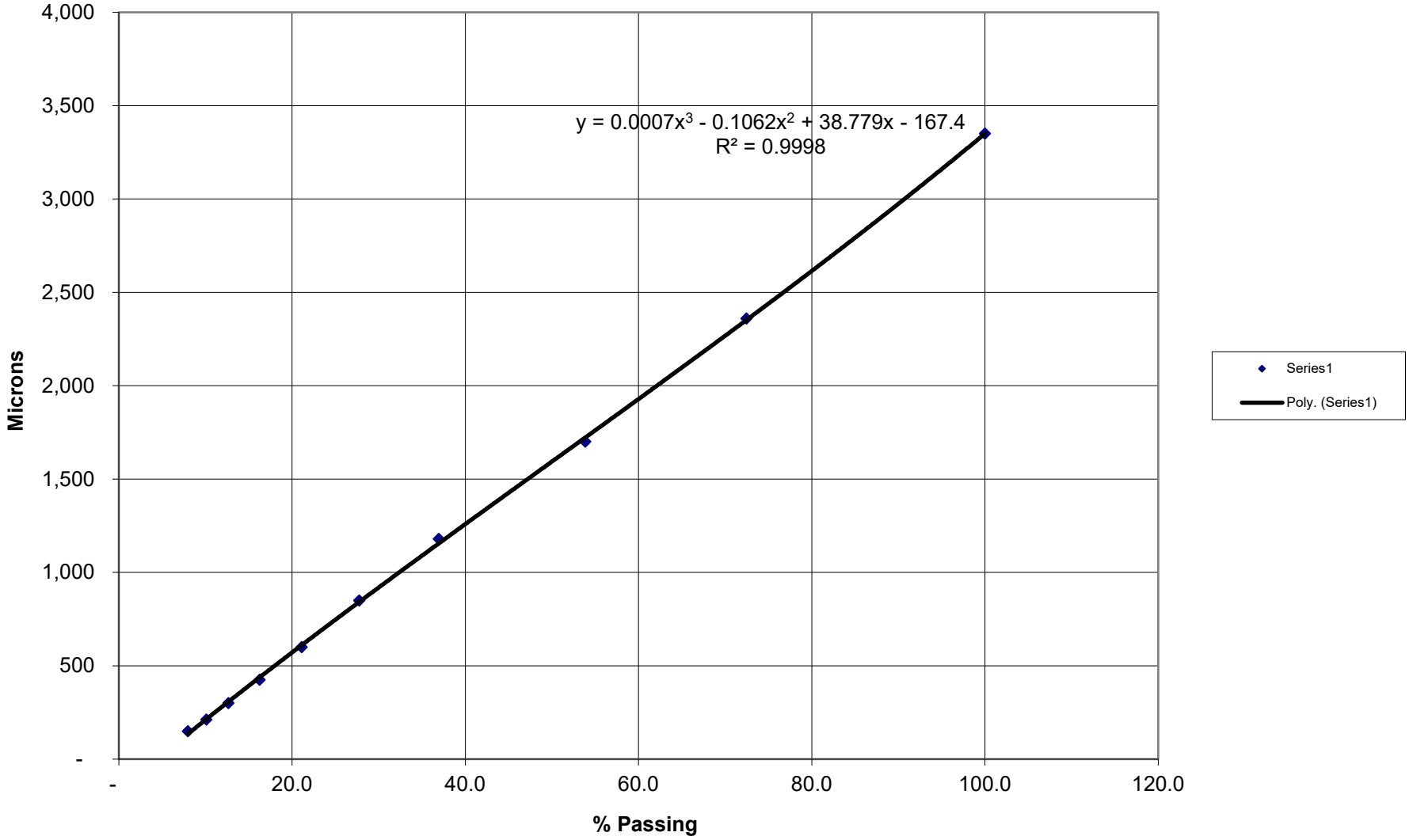
WORK INDEX - FROM GRAPH	14.07
WORK INDEX - INTERPOLATED	14.10

WORK INDEX - AVG	14.08

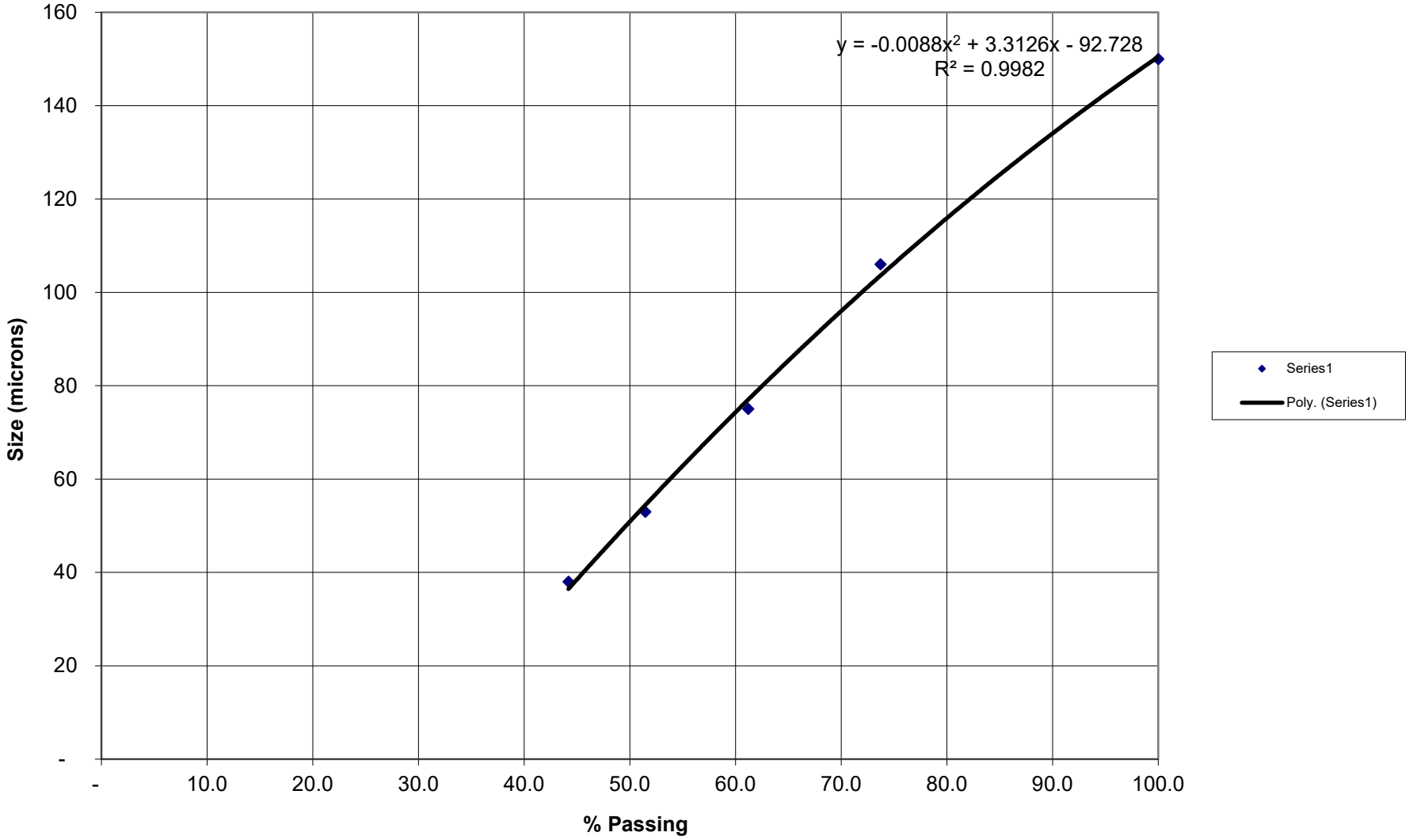
Size Analysis



Mill Feed Size Analysis



Mill Product Size Analysis



Bond Ball Mill

Feed $y = 0.0007x^3 - 0.1062x^2 + 38.779x - 167.4$

$R^2 = 0.9998$

Y	X
2613.64	80

	Microns	% Passing
6	3350	100.00
8	2360	72.47
10	1700	53.86
14	1180	36.92
20	850	27.75
28	600	21.09
35	425	16.26
48	300	12.64
100	150	7.96

Product $y = -0.0088x^2 + 3.3126x - 92.728$

$R^2 = 0.9982$

Y	X
115.96	80

Microns	% Passing
212	100.0
150	100.0
106	73.7
75	61.2
53	51.5
38	44.2

Average wt of 3 - 700 ml samples	
A1 wt (1)	1201.4
A1 wt (2)	1202.6
A1 wt (3)	1198.8
Average	1200.9

Y= A1 divided by 3.5 for 250% circulating load

X= average wt % of undersize from screening

Y=	343.1
X=	8.0

Variable	A	B	C	D	E	F	G
Calcs	average wt of 3 700 ml samples	A multiplied by X	Y minus B	C1 divided by 1.2	wt of product	E minus B	F divided by D
	A2 = E1, A3 = E2, etc.			C2 divided by G1			

		Undersize (g)				Mill Production (g)	
Cycle #	Feed (g)	In Feed	To be Produced	Mill Revs	Wt of Undersize (g)	Total Net	Per Rev
1	1200.9	95.6	247.5	211	406.6	311.0	1.476
2	406.6	32.4	310.7	210	324.1	291.7	1.386
3	324.1	25.8	317.3	229	331.2	305.4	1.334
4	331.2	26.4	316.8	237	365.4	339.0	1.428
5	365.4	29.1	314.0	220	355.4	326.3	1.484
6	355.4	28.3	314.8	212	339.3	311.0	1.465

SIZE			MILL FEED				
Tyler Mesh	Microns	Screen #1 Weight (g)	Screen #2 Weight (g)	Screen #3 Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	0.0	0.0	-	-	100.0
8	2,360	59.4	57.0	58.9	27.5	27.5	72.5
10	1,700	40.3	38.4	39.8	18.6	46.1	53.9
14	1,180	37.6	33.6	36.6	16.9	63.1	36.9
20	850	20.0	18.5	19.9	9.2	72.2	27.8
28	600	14.1	13.6	14.7	6.7	78.9	21.1
35	425	9.9	10.2	10.7	4.8	83.7	16.3
48	300	7.2	7.7	8.1	3.6	87.4	12.6
65	212	4.9	5.6	5.7	2.5	89.9	10.1
100	150	4.1	4.8	4.7	2.1	92.0	8.0
Pan		15.2	18.1	17.4	8.0	100.0	
Total		212.7	207.5	216.5	100.0		

Indirect BWI Calculations

	P_{80} Feed (microns)	P_{80} 17.5 min Grind (microns)	BWI @100 mesh
Endo/Exo Master Comp (Determined from direct BWi testing)	2656	150	15.29
Interval 2200102 Hole KXD22-03(Exo 0.35 g/mt Au - FT20)	2762	137	14.33
Interval 2200101/103 Hole KXD22-03(Exo 0.5 g/mt Au - FT21)	2727	149	15.16
Interval 2200298/667 Holes KXD22-06/09 (Endo 0.8 g/mt Au- FT22)	2773	128	13.71
Interval 2200104/596 Holes KXD22-03/08 (Endo 0.3 % Cu - FT23)	2744	140	14.55
Interval 2200291 Hole KXD22-06 (Endo 0.5 % Cu - FT24)	2601	100	11.84
Interval 2200679/014 Holes KXD22-01/09 (Endo/Exo 0.7 % Cu - FT25)	2544	84	10.66
Interval 2200127/128 Hole KXD22-03 (Endo 0.15 g/mt Au - FT26)	2731	199	18.39

APPENDIX D: GRAVITY TEST RESULTS

Gravity Summary

			Gravity Recovery				Gravity Conc. Grade			Calc. Head Grade		
Test #	Comp	Size (P_{80})	Wt	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	Cu (%)	Au (g/mt)	Ag (g/mt)	Cu (%)
EMGT1	Endo/Exo MC	65 mesh	1.3%	16.7%	9.1%	5.9%	2.05	131	2.11	0.16	19	0.47
EMGT2	Endo/Exo MC	100 mesh	0.6%	10.8%	3.9%	3.8%	2.75	156	2.83	0.16	24.5	0.46
EMGT3	Endo/Exo MC	150 mesh	0.2%	15.0%	4.1%	1.9%	14.4	534	5.58	0.15	20.7	0.47
EMGT4	Endo/Exo MC	200 mesh	0.1%	15.1%	2.1%	1.9%	29.3	479	10.0	0.17	19.5	0.44
EMGT5	Magnetic MC	65 mesh	1.3%	5.8%	1.3%	0.4%	0.71	17.3	0.21	0.16	16.9	0.65
EMGT6	Magnetic MC	100 mesh	1.6%	6.4%	5.4%	0.5%	0.61	40.6	0.20	0.15	11.9	0.67
EMGT7	Magnetic MC	150 mesh	0.7%	4.5%	3.6%	0.2%	0.87	73.7	0.20	0.13	14.1	0.67
EMGT8	Magnetic MC	200 mesh	0.2%	0.7%	0.2%	0.1%	0.64	14.2	0.24	0.14	13.6	0.68
EMGT9	High Grade Magnetic MC	65 mesh	3.5%	18.9%	6.1%	2.2%	11.4	12	0.08	2.11	6.9	0.13
EMGT10	High Grade Magnetic MC	100 mesh	2.7%	16.8%	2.4%	1.9%	14.4	13	0.08	2.32	14.9	0.11
EMGT11	High Grade Magnetic MC	150 mesh	2.4%	15.9%	8.1%	1.6%	12.5	36	0.088	1.85	10.4	0.13

RDI Project: 23009 Empire
Date: 17-Apr-23

Gravity Concentration Test 1

Purpose: To evaluate gravity separation of precious and base metals.

Sample: Approximately 2 kilograms of Endo/Exo Master Composite at P_{80} 65 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. The Gemeni concentrate was then weighed and prepared for assay of gold/silver by fire assay and base metals via ICP analysis. The gravity tails were combined for flotatoin testing.

Results:

			Assay					% Distribution				
	Weight	Weight	Au	Ag	Cu							
Product	g	%	g/mt	g/mt	%			Au	Ag	Cu		
Feed (assayed)	2,000											
Feed (calculated)	1,985	100.0	0.16	19.0	0.47			100.0	100.0	100.0		
Gravity Conc.	26.2	1.3	2.05	131	2.11			16.7	9.1	5.9		
Gravity Tail	1958.5	98.7	0.14 *	17.5 *	0.45 *			83.3	90.9	94.1		

* From calculated head grade of flotation feed

RDI Project: 23009 Empire
Date: 17-Apr-23

Gravity Concentration Test 2

Purpose: To evaluate gravity separation of precious and base metals.

Sample: Approximately 2 kilograms of Endo/Exo Master Composite at P_{80} 100 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. The Gemeni concentrate was then weighed and prepared for assay of gold/silver by fire assay and base metals via ICP analysis. The gravity tails were combined for flotatoin testing.

Results:

			Assay					% Distribution				
	Weight	Weight	Au	Ag	Cu							
Product	g	%	g/mt	g/mt	%			Au	Ag	Cu		
Feed (assayed)	2,000											
Feed (calculated)	1,985	100.0	0.16	24.5	0.46			100.0	100.0	100.0		
Gravity Conc.	12.3	0.6	2.75	156	2.83			10.8	3.9	3.8		
Gravity Tail	1972.5	99.4	0.14 *	23.7 *	0.44 *			89.2	96.1	96.2		

* From calculated head grade of flotation feed

RDI Project: 23009 Empire
Date: 17-Apr-23

Gravity Concentration Test 3

Purpose: To evaluate gravity separation of precious and base metals.

Sample: Approximately 2 kilograms of Endo/Exo Master Composite at P_{80} 150 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. The Gemeni concentrate was then weighed and prepared for assay of gold/silver by fire assay and base metals via ICP analysis. The gravity tails were combined for flotatoin testing.

Results:

			Assay					% Distribution				
	Weight	Weight	Au	Ag	Cu							
Product	g	%	g/mt	g/mt	%			Au	Ag	Cu		
Feed (assayed)	2,000											
Feed (calculated)	1,989	100.0	0.15	20.7	0.47			100.0	100.0	100.0		
Gravity Conc.	3.2	0.2	14.4	534	5.58			15.0	4.1	1.9		
Gravity Tail	1985.7	99.8	0.13 *	19.9 *	0.46 *			85.0	95.9	98.1		

* From calculated head grade of flotation feed

RDI Project: 23009 Empire
Date: 17-Apr-23

Gravity Concentration Test 4

Purpose: To evaluate gravity separation of precious and base metals.

Sample: Approximately 2 kilograms of Endo/Exo Master Composite at P_{80} 200 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. The Gemeni concentrate was then weighed and prepared for assay of gold/silver by fire assay and base metals via ICP analysis. The gravity tails were combined for flotatoin testing.

Results:

			Assay					% Distribution		
	Weight	Weight	Au	Ag	Cu					
Product	g	%	g/mt	g/mt	%			Au	Ag	Cu
Feed (assayed)	2,000									
Feed (calculated)	1,991	100.0	0.17	19.5	0.44			100.0	100.0	100.0
Gravity Conc.	1.7	0.1	29.3	479	10.00	**		15.1	2.1	1.9
Gravity Tail	1988.9	99.9	0.14	* 19.1	* 0.43	*		84.9	97.9	98.1

* From calculated head grade of flotation feed ** Estimate, not enough sample for assay

RDI Project: 23009 Empire
Date: 17-Apr-23

Gravity Concentration Test 5

Purpose: To evaluate gravity separation of precious and base metals.

Sample: Approximately 2 kilograms of Mag Master Composite at P_{80} 65 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. The Gemeni concentrate was then weighed and prepared for assay of gold/silver by fire assay and base metals via ICP analysis. The gravity tails were combined for flotatoin testing.

Results:

			Assay					% Distribution				
	Weight	Weight	Au	Ag	Cu							
Product	g	%	g/mt	g/mt	%			Au	Ag	Cu		
Feed (assayed)	2,000											
Feed (calculated)	1,980	100.0	0.16	16.9	0.65			100.0	100.0	100.0		
Gravity Conc.	25.1	1.3	0.71	17.3	0.21			5.8	1.3	0.4		
Gravity Tail	1954.6	98.7	0.15 *	16.9 *	0.66 *			94.2	98.7	99.6		

* From calculated head grade of flotation feed

RDI Project: 23009 Empire
Date: 17-Apr-23

Gravity Concentration Test 6

Purpose: To evaluate gravity separation of precious and base metals.

Sample: Approximately 2 kilograms of Mag Master Composite at P_{80} 100 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. The Gemeni concentrate was then weighed and prepared for assay of gold/silver by fire assay and base metals via ICP analysis. The gravity tails were combined for flotatoin testing.

Results:

			Assay					% Distribution				
	Weight	Weight	Au	Ag	Cu							
Product	g	%	g/mt	g/mt	%			Au	Ag	Cu		
Feed (assayed)	2,000											
Feed (calculated)	1,984	100.0	0.15	11.9	0.67			100.0	100.0	100.0		
Gravity Conc.	31.2	1.6	0.61	40.6	0.20			6.4	5.4	0.5		
Gravity Tail	1952.4	98.4	0.14 *	11.5 *	0.68 *			93.6	94.6	99.5		

* From calculated head grade of flotation feed

RDI Project: 23009 Empire
Date: 17-Apr-23

Gravity Concentration Test 7

Purpose: To evaluate gravity separation of precious and base metals.

Sample: Approximately 2 kilograms of Mag Master Composite at P_{80} 150 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. The Gemeni concentrate was then weighed and prepared for assay of gold/silver by fire assay and base metals via ICP analysis. The gravity tails were combined for flotatoin testing.

Results:

			Assay					% Distribution				
	Weight	Weight	Au	Ag	Cu							
Product	g	%	g/mt	g/mt	%			Au	Ag	Cu		
Feed (assayed)	2,000											
Feed (calculated)	1,985	100.0	0.13	14.1	0.67			100.0	100.0	100.0		
Gravity Conc.	13.8	0.7	0.87	73.7	0.20			4.5	3.6	0.2		
Gravity Tail	1971.1	99.3	0.13 *	13.7 *	0.67 *			95.5	96.4	99.8		

* From calculated head grade of flotation feed

RDI Project: 23009 Empire
Date: 17-Apr-23

Gravity Concentration Test 8

Purpose: To evaluate gravity separation of precious and base metals.

Sample: Approximately 2 kilograms of Mag Master Composite at P_{80} 200 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. The Gemeni concentrate was then weighed and prepared for assay of gold/silver by fire assay and base metals via ICP analysis. The gravity tails were combined for flotatoin testing.

Results:

			Assay					% Distribution				
	Weight	Weight	Au	Ag	Cu							
Product	g	%	g/mt	g/mt	%			Au	Ag	Cu		
Feed (assayed)	2,000											
Feed (calculated)	1,991	100.0	0.14	13.6	0.68			100.0	100.0	100.0		
Gravity Conc.	3.1	0.2	0.64	14.2	0.24			0.7	0.2	0.1		
Gravity Tail	1988.0	99.8	0.14 *	13.6 *	0.68 *			99.3	99.8	99.9		

* From calculated head grade of flotation feed

RDI Project: 23009 Empire
Date: 17-Sep-23

Gravity Concentration Test 9

Purpose: To evaluate gravity separation of precious metals.

Sample: Approximately 1 kilogram of High Grade Mag Composite at P_{80} 65 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. All gravity products were then weighed and prepared for assay of gold/silver by fire assay.

Results:

			Assay				% Distribution			
	Weight	Weight	Au	Ag	Cu					
Product	g	%	g/mt	g/mt	%		Au	Ag	Cu	
Feed (assayed)	1,000									
Feed (calculated)	996	100.0	2.11	6.9	0.13		100.0	100.0	100.0	
Knelson Conc. (calc)	115.4	11.6	6.57	6.1	0.11		36.0	10.2	9.5	
Gemeni Conc.	34.9	3.5	11.4	12.0	0.08		18.9	6.1	2.2	
Gemeni Tail	80.5	8.1	4.48	<7	0.12		17.1	4.1	7.3	
Knelson Tail	880.8	88.4	1.53	7.0	0.14		64.0	89.8	90.5	

RDI Project: 23009 Empire
Date: 17-Sep-23

Gravity Concentration Test 10

Purpose: To evaluate gravity separation of precious metals.

Sample: Approximately 1 kilogram of High Grade Mag Composite at P_{80} 100 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. All gravity products were then weighed and prepared for assay of gold/silver by fire assay.

Results:

			Assay				% Distribution			
	Weight	Weight	Au	Ag	Cu					
Product	g	%	g/mt	g/mt	%		Au	Ag	Cu	
Feed (assayed)	1,000									
Feed (calculated)	995	100.0	2.32	14.9	0.11		100.0	100.0	100.0	
Knelson Conc. (calc)	112.0	11.3	5.89	14.5	0.13		28.5	10.9	12.8	
Gemeni Conc.	27.0	2.7	14.4	13	0.08		16.8	2.4	1.9	
Gemeni Tail	85.0	8.5	3.19	15	0.14		11.7	8.6	10.8	
Knelson Tail	883.3	88.7	1.87	15	0.11		71.5	89.1	87.2	

RDI Project: 23009 Empire
Date: 17-Sep-23

Gravity Concentration Test 11

Purpose: To evaluate gravity separation of precious metals.

Sample: Approximately 1 kilogram of High Grade Mag Composite at P_{80} 150 mesh.

Procedure: The sample was slowly fed to a Knelson concentrator with water. The tailing product was collected, filtered, and dried. The concentrate collected was processed over a Model 60 Gemeni Concentration Table and two products were collected. The Gemeni concentrate and tailings were then filtered and dried. All gravity products were then weighed and prepared for assay of gold/silver by fire assay.

Results:

			Assay				% Distribution			
	Weight	Weight	Au	Ag	Cu					
Product	g	%	g/mt	g/mt	%		Au	Ag	Cu	
Feed (assayed)	1,000									
Feed (calculated)	997	100.0	1.85	10.4	0.13		100.0	100.0	100.0	
Knelson Conc. (calc)	240.7	24.1	2.70	15.2	0.10		35.2	35.3	17.8	
Gemeni Conc.	23.5	2.4	12.5	36	0.09		15.9	8.1	1.6	
Gemeni Tail	217.2	21.8	1.64	13	0.10		19.3	27.2	16.2	
Knelson Tail	756.0	75.9	1.58	8.9	0.15		64.8	64.7	82.2	

APPENDIX E: ATS TEST RESULTS

ATS Leach Summary

					8hr Extraction %			Residue. Grade			Calc. Head Grade			ATS Consumption (kg/mt)
Test #	Comp	Size (P_{80})	ATS Conc. (g/L)	Pulp Density	Au	Ag	Cu	Au (g/mt)	Ag (g/mt)	Cu (%)	Au (g/mt)	Ag (g/mt)	Cu (%)	
EMBR1	Endo/Exo MC	65 mesh	80	40%	93.7%	90.9%	43.1%	0.03	1.4	0.235	0.47	15.0	0.413	55.6
EMBR2	Endo/Exo MC	100 mesh	80	40%	93.9%	84.5%	43.0%	0.03	2.6	0.239	0.49	16.8	0.419	42.1
EMBR3	Endo/Exo MC	150 mesh	80	40%	94.7%	65.3%	42.0%	0.03	7.5	0.237	0.57	21.7	0.409	48.4
EMBR4	Endo/Exo MC	200 mesh	80	40%	92.2%	59.2%	42.1%	0.04	8.3	0.238	0.51	20.2	0.412	46.9
EMBR5	Magnetic MC	65 mesh	80	40%	92.4%	43.4%	19.7%	0.04	10.2	0.4824	0.53	18.00	0.601	49.5
EMBR6	Magnetic MC	100 mesh	80	40%	94.6%	49.9%	20.5%	0.03	7.7	0.54	0.56	15.4	0.679	49.5
EMBR7	Magnetic MC	150 mesh	80	40%	94.9%	25.6%	22.7%	0.03	21.5	0.49	0.59	28.9	0.631	47.9
EMBR8	Magnetic MC	200 mesh	80	40%	89.9%	65.7%	27.5%	0.05	5.0	0.5058	0.50	14.5	0.697	30.1
EMBR9	Endo/Exo MC	65 mesh	40	40%	85.9%	69.4%	25.7%	0.03	4.2	0.2935	0.21	13.6	0.395	38.3
EMBR10	Endo/Exo MC	65 mesh	60	40%	94.4%	85.7%	37.9%	0.02	2.2	0.2563	0.36	15.6	0.413	24.9
EMBR11	Endo/Exo MC	65 mesh	80	50%	92.4%	41.1%	38.1%	0.03	19.7	0.255	0.39	33.5	0.412	34.1
EMBR12	Magnetic MC	65 mesh	40	40%	81.2%	33.8%	10.1%	0.04	12.2	0.6	0.21	18.4	0.625	37.0
EMBR13	Magnetic MC	65 mesh	60	40%	86.0%	44.2%	16.9%	0.06	8.7	0.549	0.43	15.5	0.661	27.6
EMBR14	Magnetic MC	65 mesh	80	50%	89.4%	37.2%	21.6%	0.04	12.3	0.50	0.38	19.6	0.642	25.5

Bottle Roll Leaching Test 1

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 65 mesh	8 hours	80 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	93.7	90.9	40.4
Assayed Head, g/mt			
Calculated Head, g/mt	0.47	15.0	3938
Final Tail Assay, g/mt	0.03	1.4	2347

ATS Consumption	55.6	kg ATS/metric ton ore
Ammonium Hyroxide Added	98.0	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	88.3	67.2	30.1	29.0
2	90.4	79.4	33.1	37.1
4	90.6	83.2	37.0	40.7
8	93.7	90.9	40.4	55.6

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2619	1588	120.0	70.3	0.50			7.7	10.0	1.02
1	2614	1579		27.7		57.8		9.8	10.0	1.02
2	2619	1581				52.7		9.9		1.02
4	2619	1578				50.5		9.9		1.03
8	2616	1561				41.6		9.9		1.04
Total			120.0	98.0						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.47		15.0		3938		
1 hour Preg		1579		0.26		6.33		855	30
2 hour Preg		1581		0.26		7.35		916	30
4 hour Preg		1578		0.26		7.58		997	30
8 hour Preg		1561		0.27		8.25		1075	
8 hour Dry Residue	989.7		0.03		1.4		2347		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 2

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	93.9	84.5	40.3
Assayed Head, g/mt			
Calculated Head, g/mt	0.49	16.8	4004
Final Tail Assay, g/mt	0.03	2.6	2391

ATS Consumption	42.1	kg ATS/metric ton ore
Ammonium Hyroxide Added	103.4	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	80.8	23.7	33.4	21.4
2	89.1	69.8	36.4	28.8
4	90.3	76.6	38.3	33.1
8	93.9	84.5	40.3	42.1

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2620	1588	120.0	51.7	0.50			7.9	10.0	1.02
1	2619	1583		35.6		62.4		9.7	10.0	1.02
2	2619	1583				57.8		9.9		1.02
4	2619	1580		16.1		55.2		9.9	10.0	1.03
8	2619	1586				49.3		9.9		1.02
Total			120.0	103.4						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.49		16.8		4004		
1 hour Preg		1583		0.25		2.49		950	30
2 hour Preg		1583		0.27		7.31		1010	30
4 hour Preg		1580		0.27		7.90		1042	30
8 hour Preg		1586		0.28		8.55		1071	
8 hour Dry Residue	991.9		0.03		2.6		2391		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 3

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 150 mesh	8 hours	80 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	94.7	65.3	39.2
Assayed Head, g/mt			
Calculated Head, g/mt	0.57	21.7	3901
Final Tail Assay, g/mt	0.03	7.5	2372

ATS Consumption	48.4	kg ATS/metric ton ore
Ammonium Hyroxide Added	98.6	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	72.9	2.1	30.5	29.0
2	72.3	7.4	34.0	41.7
4	84.1	27.6	35.5	47.9
8	94.7	65.3	39.2	48.4

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2620	1588	120.0	46.7	0.50			8.1	10.0	1.02
1	2620	1577		36.8		57.8		9.7	9.9	1.03
2	2621	1589				49.4		9.9		1.02
4	2617	1585		15.1		45.6		9.9	10.0	1.02
8	2618	1574				45.6		9.9		1.03
Total			120.0	98.6						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.57		21.7		3901		
1 hour Preg		1577		0.26		0.29		863	30
2 hour Preg		1589		0.25		1.00		930	30
4 hour Preg		1585		0.29		3.74		953	30
8 hour Preg		1574		0.33		8.87		1034	
8 hour Dry Residue	995.6		0.03		7.5		2372		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 4

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 200 mesh	8 hours	80 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	92.2	59.2	39.4
Assayed Head, g/mt			
Calculated Head, g/mt	0.51	20.2	3932
Final Tail Assay, g/mt	0.04	8.3	2384

ATS Consumption	46.9	kg ATS/metric ton ore
Ammonium Hyroxide Added	101.9	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	95.7	0.4	31.5	29.0
2	83.5	0.8	35.2	32.5
4	86.5	2.3	34.6	37.3
8	92.2	59.2	39.4	46.9

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2620	1588	120.0	47.1	0.50			8.0	10.0	1.02
1	2620	1576		38.1		57.8		9.6	10.0	1.03
2	2623	1586				55.2		9.9		1.02
4	2621	1571		16.7		52.7		9.9	9.9	1.03
8	2621	1545				47.4		9.9		1.05
Total			120.0	101.9						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.51		20.2		3932		
1 hour Preg		1576		0.31		0.05		897	30
2 hour Preg		1586		0.26		0.10		967	30
4 hour Preg		1571		0.27		0.29		945	30
8 hour Preg		1545		0.29		7.72		1064	
8 hour Dry Residue	997.5		0.04		8.3		2384		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 5

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 65 mesh	8 hours	80 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	92.4	43.4	17.1
Assayed Head, g/mt			
Calculated Head, g/mt	0.53	18.0	5818
Final Tail Assay, g/mt	0.04	10.2	4824

ATS Consumption	49.5	kg ATS/metric ton ore
Ammonium Hyroxide Added	111.5	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	93.0	17.9	15.8	24.4
2	90.5	36.5	17.1	36.8
4	93.4	39.1	17.5	32.8
8	92.4	43.4	17.1	49.5

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2619	1587	120.0	47.8	0.50			7.6	10.1	1.02
1	2614	1575		35.7		60.8		9.6	10.0	1.03
2	2618	1554				53.7		9.9		1.04
4	2617	1584		28.0		55.2		9.8	10.0	1.02
8	2620	1585				44.7		10.0		1.02
Total			120.0	111.5						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.53		18.0		5818		
1 hour Preg		1575		0.31		2.03		692	30
2 hour Preg		1554		0.30		4.16		737	30
4 hour Preg		1584		0.30		4.29		728	30
8 hour Preg		1585		0.29		4.70		701	
8 hour Dry Residue	992.5		0.04		10.2		4824		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 6

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	94.6	49.9	18.2
Assayed Head, g/mt			
Calculated Head, g/mt	0.56	15.4	6603
Final Tail Assay, g/mt	0.03	7.7	5402

ATS Consumption	49.5	kg ATS/metric ton ore
Ammonium Hyroxide Added	106.6	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	96.3	14.4	12.6	21.9
2	95.5	39.8	15.4	27.7
4	98.9	45.8	17.2	29.0
8	94.6	49.9	18.2	49.5

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2619	1587	120.0	47.8	0.50			7.7	10.1	1.02
1	2617	1575		35.3		62.4		9.6	10.0	1.03
2	2621	1560				59.3		9.9		1.04
4	2618	1577		23.5		57.8		9.8	9.9	1.03
8	2619	1585				44.7		9.9		1.02
Total			120.0	106.6						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.56		15.4		6603		
1 hour Preg		1575		0.34		1.40		636	30
2 hour Preg		1560		0.33		3.88		750	30
4 hour Preg		1577		0.34		4.34		805	30
8 hour Preg		1585		0.31		4.64		831	
8 hour Dry Residue	993.6		0.03		7.7		5402		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 7

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 150 mesh	8 hours	80 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	94.9	25.6	20.3
Assayed Head, g/mt			
Calculated Head, g/mt	0.59	28.9	6121
Final Tail Assay, g/mt	0.03	21.5	4878

ATS Consumption	47.9	kg ATS/metric ton ore
Ammonium Hyroxide Added	85.5	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	85.1	6.6	17.3	23.2
2	87.9	18.9	17.7	25.2
4	94.6	23.2	17.6	36.6
8	94.9	25.6	20.3	47.9

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2637	1605	120.0	47.9	0.50			7.8	10.0	1.02
1	2637	1594		37.6		60.8		9.8	10.0	1.03
2	2642	1580				60.0		10.0		1.04
4	2623	1583				52.7		10.0		1.03
8	2619	1583				45.6		9.9		1.02
Total			120.0	85.5						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.59		28.9		6121		
1 hour Preg		1594		0.31		1.19		776	30
2 hour Preg		1580		0.32		3.42		783	30
4 hour Preg		1583		0.34		4.14		766	30
8 hour Preg		1583		0.33		4.50		857	
8 hour Dry Residue	997.6		0.03		21.5		4878		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 8

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 200 mesh	8 hours	80 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	89.9	65.7	25.4
Assayed Head, g/mt			
Calculated Head, g/mt	0.50	14.5	6784
Final Tail Assay, g/mt	0.05	5.0	5058

ATS Consumption	30.1	kg ATS/metric ton ore
Ammonium Hyroxide Added	83.6	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	91.7	2.4	18.4	3.1
2	82.9	9.1	21.3	27.2
4	86.0	27.7	20.7	29.2
8	89.9	65.7	25.4	30.1

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2618	1587	120.0	47.3	0.50			7.9	9.9	1.02
1	2618	1578		36.3		74.1		9.7	9.9	1.03
2	2621	1567				59.3		9.9		1.04
4	2620	1571				57.8		9.9		1.03
8	2620	1591				56.5		9.9		1.02
Total			120.0	83.6						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.50		14.5		6784		
1 hour Preg		1578		0.29		0.22		904	30
2 hour Preg		1567		0.26		0.84		1020	30
4 hour Preg		1571		0.26		2.53		971	30
8 hour Preg		1591		0.26		5.91		1147	
8 hour Dry Residue	998.2		0.05		5.0		5058		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 9

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 40 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 65 mesh	8 hours	40.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	85.9	69.4	21.9
Assayed Head, g/mt			
Calculated Head, g/mt	0.21	13.6	3758
Final Tail Assay, g/mt	0.03	4.2	2935

ATS Consumption	38.3	kg ATS/metric ton ore
Ammonium Hyroxide Added	86.1	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	56.5	65.2	22.8	27.0
2	71.4	72.7	25.4	35.7
4	86.0	75.8	24.4	37.2
8	85.9	69.4	21.9	38.3

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2567	1552	60.0	86.1	0.50			8.2	10.1	1.01
1	2567	1551				21.4		10.1		1.01
2	2565	1549				15.8		10.0		1.01
4	2552	1536				15.0		10.0		1.01
8	2551	1546				14.2		10.0		1.00
Total			60.0	86.1						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.21		13.6		3758		
1 hour Preg		1551		0.08		5.67		663	30
2 hour Preg		1549		0.10		6.22		718	30
4 hour Preg		1536		0.12		6.42		686	30
8 hour Preg		1546		0.11		5.70		611	
8 hour Dry Residue	993.7		0.03		4.2		2935		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 10

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 60 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 65 mesh	8 hours	60.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	94.4	85.7	34.9
Assayed Head, g/mt			
Calculated Head, g/mt	0.36	15.6	3938
Final Tail Assay, g/mt	0.02	2.2	2563

ATS Consumption	24.9	kg ATS/metric ton ore
Ammonium Hyroxide Added	99.6	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	83.0	64.2	30.5	23.5
2	88.7	75.4	33.5	26.1
4	95.1	83.5	39.4	26.1
8	94.4	85.7	34.9	24.9

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2577	1546	90.0	99.6	0.50			8.0	10.1	1.02
1	2577	1547				43.1		10.0		1.02
2	2568	1542				41.6		10.0		1.02
4	2557	1542				41.6		10.0		1.01
8	2556	1544				42.3		10.1		1.01
Total			90.0	99.6						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.36		15.6		3938		
1 hour Preg		1547		0.19		6.41		886	30
2 hour Preg		1542		0.20		7.42		950	30
4 hour Preg		1542		0.21		8.09		1082	30
8 hour Preg		1544		0.21		8.14		949	
8 hour Dry Residue	991.5		0.02		2.2		2563		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 11

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 65 mesh	8 hours	80.0 g/L	49% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	92.4	41.1	35.0
Assayed Head, g/mt			
Calculated Head, g/mt	0.39	33.5	3921
Final Tail Assay, g/mt	0.03	19.7	2550

ATS Consumption	34.1	kg ATS/metric ton ore
Ammonium Hyroxide Added	56.1	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	79.8	23.8	28.6	28.4
2	98.3	34.9	31.0	26.7
4	91.3	38.6	37.1	31.0
8	92.4	41.1	35.0	34.1

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2041	1011	80.0	56.1	0.50			8.0	10.0	1.03
1	2041	1004				51.6		10.0		1.04
2	2041	1015				52.7		9.9		1.03
4	2038	1017				48.4		9.9		1.02
8	2035	1013				45.6		9.9		1.02
Total			80.0	56.1						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.39		33.5		3921		
1 hour Preg		1004		0.31		7.85		1287	30
2 hour Preg		1015		0.37		11.20		1334	30
4 hour Preg		1017		0.33		12.02		1529	30
8 hour Preg		1013		0.33		12.55		1413	
8 hour Dry Residue	992.3		0.03		19.7		2550		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 12

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 40 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 65 mesh	8 hours	40.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	81.2	33.8	7.3
Assayed Head, g/mt			
Calculated Head, g/mt	0.21	18.4	6059
Final Tail Assay, g/mt	0.04	12.2	5617

ATS Consumption	37.0	kg ATS/metric ton ore
Ammonium Hyroxide Added	88.1	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	66.1	26.6	9.4	25.6
2	83.7	29.8	8.4	39.3
4	149.4	34.4	13.6	39.1
8	81.2	33.8	7.3	37.0

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2568	1537	60.0	88.1	0.50			7.7	10.1	1.02
1	2568	1555				22.2		10.0		1.01
2	2570	1545				13.5		10.0		1.02
4	2552	1537				13.7		10.0		1.01
8	2556	1552				14.9		10.0		1.00
Total			60.0	88.1						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.21		18.4		6059		
1 hour Preg		1555		0.09		3.15		482	30
2 hour Preg		1545		0.11		3.48		436	30
4 hour Preg		1537		0.20		3.98		636	30
8 hour Preg		1552		0.10		3.80		377	
8 hour Dry Residue	995.6		0.04		12.2		5617		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 13

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 60 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 65 mesh	8 hours	60.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	86.0	44.2	14.5
Assayed Head, g/mt			
Calculated Head, g/mt	0.43	15.5	6421
Final Tail Assay, g/mt	0.06	8.7	5490

ATS Consumption	27.6	kg ATS/metric ton ore
Ammonium Hyroxide Added	105.5	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	61.9	28.6	14.2	18.1
2	66.5	39.4	14.2	29.5
4	128.6	71.2	9.4	28.9
8	86.0	44.2	14.5	27.6

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2575	1544	90.0	105.5	0.50			7.7	10.1	1.02
1	2575	1549				46.5		10.1		1.02
2	2575	1536				39.5		10.1		1.03
4	2557	1524				40.2		10.0		1.02
8	2554	1528				40.9		10.0		1.02
Total			90.0	105.5						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.43		15.5		6421		
1 hour Preg		1549		0.17		2.85		703	30
2 hour Preg		1536		0.18		3.90		697	30
4 hour Preg		1524		0.35		7.08		489	30
8 hour Preg		1528		0.23		4.19		692	
8 hour Dry Residue	995.6		0.06		8.7		5490		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 14

RDi Project: 23009 Empire
Date: 25-Apr-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 65 mesh	8 hours	80.0 g/L	49% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	89.4	37.2	19.1
Assayed Head, g/mt			
Calculated Head, g/mt	0.38	19.6	6222
Final Tail Assay, g/mt	0.04	12.3	5032

ATS Consumption	25.5	kg ATS/metric ton ore
Ammonium Hyroxide Added	59.2	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	84.8	24.9	14.7	29.5
2	91.0	34.4	18.5	25.7
4	39.2	21.9	18.4	25.6
8	89.4	37.2	19.1	25.5

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2043	1022	80.0	59.2	0.50			7.7	10.0	1.02
1	2043	1007				50.5		9.9		1.04
2	2041	1014				53.9		9.9		1.03
4	2037	1017				53.9		9.9		1.02
8	2040	1017				53.9		9.9		1.02
Total			80.0	59.2						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.38		19.6		6222		
1 hour Preg		1007		0.31		4.77		1070	30
2 hour Preg		1014		0.32		6.40		1267	30
4 hour Preg		1017		0.12		3.83		1227	30
8 hour Preg		1017		0.30		6.61		1239	
8 hour Dry Residue	986.0		0.04		12.3		5032		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 23

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	91.9	80.5	41.2
Assayed Head, g/mt			
Calculated Head, g/mt	0.62	19.0	4685
Final Tail Assay, g/mt	0.05	3.7	2757

ATS Consumption	51.2	kg ATS/metric ton ore
Ammonium Hyroxide Added	105.9	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	75.9	56.2	30.2	54.1
2	86.0	69.3	33.8	44.1
4	82.1	73.9	36.9	58.2
8	91.9	80.5	41.2	51.2

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2549	1518	120.0	96.3	0.50			8.1	9.9	1.02
1	2548	1474				44.7		9.8		1.05
2	2546	1504				50.5		9.8		1.03
4	2546	1486		9.60		41.6		9.7	9.9	1.04
8	2544	1479				46.5		9.8		1.04
Total			120.0	105.9						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.62		19.0		4685		
1 hour Preg		1474		0.32		7.23		1079	30
2 hour Preg		1504		0.35		8.60		1152	30
4 hour Preg		1486		0.33		9.11		1242	30
8 hour Preg		1479		0.37		9.82		1358	
8 hour Dry Residue	1000.0		0.05		3.7		2757		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 24

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper at various particle sizes with ammonium thiosulfate.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	89.6	39.3	17.1
Assayed Head, g/mt			
Calculated Head, g/mt	0.58	23.4	7464
Final Tail Assay, g/mt	0.06	14.2	6187

ATS Consumption	62.0	kg ATS/metric ton ore
Ammonium Hyroxide Added	107.2	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	79.1	29.4	15.1	36.1
2	86.8	35.3	15.9	43.8
4	86.6	40.2	16.7	57.6
8	89.6	39.3	17.1	62.0

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	Reagents Added, g NH_4OH	Reagents Added, g $CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	pH Adjust	PLS SG
0	2544	1513	120.0	98.7	0.50			7.7	10.0	1.02
1	2544	1484				56.5		9.8		1.04
2	2544	1478				51.6		9.8		1.05
4	2544	1499		8.50		41.6		9.7	9.9	1.03
8	2546	1490				38.9		9.8		1.04
Total			120.0	107.2						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.58		23.4		7464		
1 hour Preg		1484		0.31		4.63		878	30
2 hour Preg		1478		0.33		5.49		908	30
4 hour Preg		1499		0.32		6.07		920	30
8 hour Preg		1490		0.33		5.85		928	
8 hour Dry Residue	1000.0		0.06		14.2		6187		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #1 - Cycle 1

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	92.7	76.3	43.0
Assayed Head, g/mt			
Calculated Head, g/mt	0.55	20.5	4670
Final Tail Assay, g/mt	0.04	4.9	2664

ATS Consumption	43.6	kg ATS/metric ton ore
Ammonium Hyroxide Added	108.1	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	85.9	38.9	33.6	48.8
2	90.0	58.7	35.9	47.3
4	95.1	69.5	38.7	49.0
8	92.7	76.3	43.0	43.6

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2546	1486	120.0	102.4	0.50			8.2	9.9	1.04
1	2546	1486				48.4		9.8		1.04
2	2545	1486				49.4		9.8		1.04
4	2546	1481		5.70		48.4		9.7	9.9	1.04
8	2547	1493				51.6		9.9		1.04
Total			120.0	108.1						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.55		20.5		4670		
1 hour Preg		1486		0.31		5.30		1041	30
2 hour Preg		1486		0.32		7.88		1091	30
4 hour Preg		1481		0.34		9.22		1160	30
8 hour Preg		1493		0.32		9.89		1258	
8 hour Dry Residue	985.6		0.04		4.9		2664		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #1 - Cycle 2

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	92.1	74.6	25.4
Assayed Head, g/mt			
Calculated Head, g/mt	0.38	19.8	3661
Final Tail Assay, g/mt	0.03	5.0	2730

ATS Consumption	42.3	kg ATS/metric ton ore
Ammonium Hyroxide Added	125.9	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	97.9	60.2	23.2	32.2
2	109.2	72.0	25.6	51.5
4	97.8	72.1	24.9	48.4
8	92.1	74.6	25.4	42.3

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2532	1470	78.9	120.2	0.50	27.4		8.8	9.6	1.04
1	2531	1480				59.2		9.5	9.8	1.03
2	2530	1477				46.5		9.6	9.9	1.04
4	2528	1481		5.70		48.4		9.8		1.03
8	2527	1474				52.7		9.8		1.04
Total			78.9	125.9						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.38		19.8		3661		
1 hour Preg		1480		0.49		9.10		896	30
2 hour Preg		1477		0.51		10.50		944	30
4 hour Preg		1481		0.48		10.30		915	30
8 hour Preg		1474		0.46		10.50		918	
8 hour Dry Residue	980.9		0.03		5.0		2730		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #1 - Cycle 3

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	91.9	82.3	41.2
Assayed Head, g/mt			
Calculated Head, g/mt	0.37	20.8	4747
Final Tail Assay, g/mt	0.03	3.7	2793

ATS Consumption	48.0	kg ATS/metric ton ore
Ammonium Hyroxide Added	109.3	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	71.8	61.4	37.2	22.6
2	89.9	76.2	35.8	32.9
4	77.0	74.1	39.7	40.3
8	91.9	82.3	41.2	48.0

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	Reagents Added, g NH_4OH	Reagents Added, g $CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	pH Adjust	PLS SG
0	2563	1502	64.1	82.8	0.50	39.5		8.7	9.9	1.04
1	2563	1492		26.5		67.8		9.7	9.9	1.05
2	2557	1496				60.8		9.9		1.04
4	2543	1481				56.5		9.8		1.04
8	2534	1473				51.6		9.8		1.04
Total			64.1	109.3						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.37		20.8		4747		
1 hour Preg		1492		0.51		9.46		1741	30
2 hour Preg		1496		0.55		11.30		1669	30
4 hour Preg		1481		0.52		10.90		1784	30
8 hour Preg		1473		0.55		11.90		1811	
8 hour Dry Residue	986.7		0.03		3.7		2793		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #1 - Cycle 4

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	~~92.1~~	64.3	41.3
Assayed Head, g/mt			
Calculated Head, g/mt	~~0.38~~	20.4	4623
Final Tail Assay, g/mt	~~0.03~~	7.3	2715

ATS Consumption	31.7	kg ATS/metric ton ore
Ammonium Hyroxide Added	67.8	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	100.3	54.3	35.3	31.5
2	110.3	54.5	34.9	24.9
4	97.6	60.9	39.5	28.7
8	92.1	64.3	41.3	31.7

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2580	1517	42.0	67.8	0.50	55.2		9.4	10.0	1.04
1	2580	1517				62.4		9.9		1.04
2	2555	1494				67.7		9.9		1.04
4	2538	1478				65.9		9.9		1.04
8	2537	1474				64.1		9.8		1.04
Total			42.0	67.8						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.38		20.4		4623		
1 hour Preg		1517		~~0.49~~		8.20		1859	30
2 hour Preg		1494		~~0.51~~		8.19		1841	30
4 hour Preg		1478		~~0.48~~		8.99		1974	30
8 hour Preg		1474		~~0.46~~		9.32		2009	
8 hour Dry Residue	987.7		0.03		7.3		2715		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #1 - Cycle 5

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	94.6	64.4	16.4
Assayed Head, g/mt			
Calculated Head, g/mt	0.37	16.6	3086
Final Tail Assay, g/mt	0.02	5.9	2578

ATS Consumption	26.1	kg ATS/metric ton ore
Ammonium Hyroxide Added	95.2	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	76.4	55.4	13.7	14.6
2	54.0	36.2	6.6	49.3
4	88.9	68.5	19.9	23.6
8	94.6	64.4	16.4	26.1

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2558	1487	52.6	95.2	0.50	47.4		8.7	9.9	1.05
1	2559	1468				74.1		9.8		1.06
2	2560	1006				74.1		9.9		1.55
4	2558	1474				67.8		9.9		1.06
8	2557	1478				65.9		9.8		1.05
Total			52.6	95.2						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.37		16.6		3086		
1 hour Preg		1468		0.77		7.15		965	30
2 hour Preg		1006		0.77		6.69		871	30
4 hour Preg		1474		0.79		8.34		1082	30
8 hour Preg		1478		0.80		7.71		1001	
8 hour Dry Residue	983.3		0.02		5.9		2578		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #1 - Cycle 6

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Endo/Exo Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	93.0	70.4	13.0
Assayed Head, g/mt			
Calculated Head, g/mt	0.28	12.2	3165
Final Tail Assay, g/mt	0.02	3.6	2753

ATS Consumption	20.2	kg ATS/metric ton ore
Ammonium Hyroxide Added	79.9	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	83.4	62.1	13.5	12.3
2	88.7	61.8	13.5	10.6
4	52.9	62.9	13.0	36.7
8	93.0	70.4	13.0	20.2

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2581	1485	35.7	79.9	0.50	64.1		8.9	9.9	1.06
1	2580	1502				79.1		9.9		1.05
2	2557	1473				81.8		9.8		1.06
4	2584	1519				62.4		9.8		1.04
8	2578	1499				74.1		9.8		1.05
Total			35.7	79.9						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.28		12.2		3165		
1 hour Preg		1502		0.70		5.94		852	30
2 hour Preg		1473		0.71		5.91		852	30
4 hour Preg		1519		0.64		5.76		827	30
8 hour Preg		1499		0.71		6.32		827	
8 hour Dry Residue	983.3		0.02		3.6		2753		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #2 - Cycle 1

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	89.7	41.4	20.4
Assayed Head, g/mt			
Calculated Head, g/mt	0.58	22.3	7211
Final Tail Assay, g/mt	0.06	13.1	5738

ATS Consumption	56.4	kg ATS/metric ton ore
Ammonium Hyroxide Added	117.3	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	80.0	31.8	18.5	39.3
2	87.0	38.3	20.2	41.8
4	90.2	42.7	19.9	50.4
8	89.7	41.4	20.4	56.4

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2542	1483	120.0	109.3	0.50			7.9	9.9	1.04
1	2543	1472				55.2		9.9		1.05
2	2542	1494				52.7		9.8		1.03
4	2543	1483		8.00		47.4		9.7	9.9	1.04
8	2544	1493				43.1		9.9		1.03
Total			120.0	117.3						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.58		22.3		7211		
1 hour Preg		1472		0.31		4.77		896	30
2 hour Preg		1494		0.33		5.55		944	30
4 hour Preg		1483		0.34		6.14		915	30
8 hour Preg		1493		0.32		5.78		918	
8 hour Dry Residue	986.6		0.06		13.1		5738		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #2 - Cycle 2

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	89.4	25.8	20.3
Assayed Head, g/mt			
Calculated Head, g/mt	0.47	37.1	7237
Final Tail Assay, g/mt	0.05	27.5	5767

ATS Consumption	33.0	kg ATS/metric ton ore
Ammonium Hyroxide Added	180.7	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	93.8	22.7	17.2	25.0
2	97.1	22.7	19.6	42.3
4	88.7	23.6	19.7	50.6
8	89.4	25.8	20.3	33.0

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2532	1473	85.8	118.4	0.50	22.8		8.4	9.6	1.04
1	2532	1479		31.4		64.1		9.5	9.8	1.04
2	2529	1475		30.9		52.7		9.6	9.9	1.04
4	2529	1466				47.4		9.8		1.04
8	2529	1465				59.3		9.8		1.04
Total			85.8	180.7						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.47		37.1		7237		
1 hour Preg		1479		0.56		7.13		1135	30
2 hour Preg		1475		0.57		7.02		1239	30
4 hour Preg		1466		0.54		7.17		1231	30
8 hour Preg		1465		0.53		7.62		1241	
8 hour Dry Residue	985.8		0.05		27.5		5767		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #2 - Cycle 3

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	80.9	48.4	18.7
Assayed Head, g/mt			
Calculated Head, g/mt	0.31	18.4	7407
Final Tail Assay, g/mt	0.06	9.5	6018

ATS Consumption	49.9	kg ATS/metric ton ore
Ammonium Hyroxide Added	89.2	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	74.3	41.4	14.9	18.4
2	66.5	44.0	15.5	32.1
4	72.1	47.4	16.9	38.4
8	80.9	48.4	18.7	49.9

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2606	1536	46.5	69.6	0.50	49.4		8.0	9.9	1.05
1	2597	1538		19.6		67.8		9.7	9.9	1.04
2	2583	1529				59.3		9.9		1.04
4	2555	1495				56.5		9.9		1.04
8	2539	1481				49.4		9.8		1.04
Total			46.5	89.2						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.31		18.4		7407		
1 hour Preg		1538		0.56		5.50		1243	30
2 hour Preg		1529		0.54		5.74		1263	30
4 hour Preg		1495		0.55		6.17		1335	30
8 hour Preg		1481		0.57		6.23		1417	
8 hour Dry Residue	986.9		0.06		9.5		6018		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #2 - Cycle 4

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	38% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	~~91.4~~	41.3	20.3
Assayed Head, g/mt			
Calculated Head, g/mt	~~0.46~~	17.9	7448
Final Tail Assay, g/mt	~~0.04~~	10.5	5933

ATS Consumption	32.6	kg ATS/metric ton ore
Ammonium Hyroxide Added	86.9	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	99.3	31.8	14.0	9.1
2	100.5	38.9	18.0	27.9
4	90.6	39.4	17.5	24.1
8	91.4	41.3	20.3	32.6

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2602	1515	45.0	86.9	0.50	53.9		8.1	9.9	1.06
1	2602	1538				76.5		9.9		1.04
2	2571	1503				65.9		9.8		1.05
4	2541	1473				69.8		9.8		1.05
8	2542	1473				64.1		9.8		1.05
Total			45.0	86.9						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.46		17.9		7448		
1 hour Preg		1538		~~0.56~~		4.65		1502	30
2 hour Preg		1503		~~0.57~~		5.50		1700	30
4 hour Preg		1473		~~0.54~~		5.56		1677	30
8 hour Preg		1473		~~0.53~~		5.69		1801	
8 hour Dry Residue	989.0		0.04		10.5		5933		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #2 - Cycle 5

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	70.2	31.4	4.0
Assayed Head, g/mt			
Calculated Head, g/mt	0.34	15.6	5999
Final Tail Assay, g/mt	0.10	10.7	5761

ATS Consumption	31.7	kg ATS/metric ton ore
Ammonium Hyroxide Added	90.2	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	61.4	31.0	2.8	12.3
2	65.9	29.7	4.2	12.1
4	69.1	35.2	4.5	24.3
8	70.2	31.4	4.0	31.7

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2558	1481	29.7	90.2	0.50	62.4		8.8	9.9	1.05
1	2558	1484				74.1		9.9		1.05
2	2557	1487				74.1		9.8		1.05
4	2557	1489				65.9		9.8		1.05
8	2558	1494				60.8		9.8		1.04
Total			29.7	90.2						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.34		15.6		5999		
1 hour Preg		1484		0.71		4.21		779	30
2 hour Preg		1487		0.72		4.01		830	30
4 hour Preg		1489		0.72		4.51		840	30
8 hour Preg		1494		0.72		4.04		815	
8 hour Dry Residue	986.8		0.10		10.7		5761		

(1) Based on calculated head assays.

Bottle Roll Leaching Locked Cycle Test #2 - Cycle 6

RDi Project: 23009 Empire
Date: 15-May-23

Purpose: To examine the extraction of gold, silver, and copper with recycle of ammonium thiosulfate PLS.

Sample: Approximately 1000 grams of Mag Master composite.

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	39% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	91.3	45.2	7.4
Assayed Head, g/mt			
Calculated Head, g/mt	0.46	12.0	6531
Final Tail Assay, g/mt	0.04	6.6	6048

ATS Consumption	17.8	kg ATS/metric ton ore
Ammonium Hyroxide Added	87.4	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	82.7	41.7	7.3	10.3
2	79.7	40.9	6.5	6.2
4	87.2	45.3	7.6	9.8
8	91.3	45.2	7.4	17.8

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2581	1493	53.2	87.4	0.50	50.5		8.7	9.9	1.06
1	2581	1497				79.1		9.8		1.06
2	2577	1497				81.8		9.8		1.05
4	2578	1503				79.1		9.8		1.05
8	2578	1498				74.1		9.8		1.05
Total			53.2	87.4						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.46		12.0		6531		
1 hour Preg		1497		0.74		4.31		782	30
2 hour Preg		1497		0.72		4.18		740	30
4 hour Preg		1503		0.74		4.45		782	30
8 hour Preg		1498		0.75		4.39		766	
8 hour Dry Residue	986.8		0.04		6.6		6048		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 27

RDi Project: 23009 Empire
Date: 17-Jul-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 1000 grams of FT17 tails (Endo/Exo Master composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 1, 2 and 4 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	89.7	68.5	51.0
Assayed Head, g/mt			
Calculated Head, g/mt	0.58	13.5	3972
Final Tail Assay, g/mt	0.06	4.3	1946

ATS Consumption	51.6	kg ATS/metric ton ore
Ammonium Hyroxide Added	93.1	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
1	79.4	51.3	46.2	55.7
2	87.9	62.2	51.9	51.3
4	86.2	69.6	44.0	51.6
8	89.7	68.5	51.0	51.6

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	2501	1472	120.0	93.1	0.50			7.7	10.1	1.02
1	2501	1466				44.7		10.1		1.02
2	2501	1474				47.4		10.1		1.02
4	2500	1468				47.4		10.0		1.02
8	2500	1468				47.4		10.1		1.02
Total			120.0	93.1						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	1000								
Feed (computed)			0.58		13.5		3972		
1 hour Preg		1466		0.31		4.63		1223	30
2 hour Preg		1474		0.33		5.49		1341	30
4 hour Preg		1468		0.32		6.07		1112	30
8 hour Preg		1468		0.33		5.85		1274	
8 hour Dry Residue	977.8		0.06		4.3		1946		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 37

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of FT20 tails (Endo/Exo varability composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	69.8	54.7	24.8
Assayed Head, g/mt			
Calculated Head, g/mt	0.36	7.7	4332
Final Tail Assay, g/mt	0.11	<7	3258

ATS Consumption	50.2	kg ATS/metric ton ore
Ammonium Hyroxide Added	50.9	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	72.1	49.5	23.1	39.3
4	73.5	52.2	33.6	46.1
6	72.5	53.9	31.1	49.0
8	69.8	54.7	24.8	50.2

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	1751	1030	84.0	35.6	0.35			7.9	10.0	1.02
2	1751	1023				55.2		9.9		1.03
4	1750	1024				50.5		9.9		1.03
6	1752	1028				48.4		9.8		1.02
8	1754	1030				47.4		9.8		1.02
Total			84.0	35.6						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.36		7.7		4332		
2 hour Preg		1023		0.18		2.58		795	30
4 hour Preg		1024		0.18		2.64		1083	30
6 hour Preg		1028		0.17		2.64		977	30
8 hour Preg		1030		0.16		2.60		768	
8 hour Dry Residue	689.5		0.11		<7		3258		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 38

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of FT21 tails (Endo/Exo varability composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	36.6	41.6	25.1
Assayed Head, g/mt			
Calculated Head, g/mt	0.74	6.0	3193
Final Tail Assay, g/mt	0.47	<7	2393

ATS Consumption	48.8	kg ATS/metric ton ore
Ammonium Hyroxide Added	55.6	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	37.5	35.4	19.9	50.5
4	37.4	38.0	25.6	49.0
6	36.4	40.0	24.6	50.5
8	36.6	41.6	25.1	48.8

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	1751	1030	84.0	38.9	0.35			7.2	10.0	1.02
2	1751	1026				47.4		9.9		1.02
4	1752	1027				48.4		9.9		1.02
6	1751	1026				47.4		9.9		1.02
8	1755	1030				48.4		9.9		1.02
Total			84.0	38.9						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.74		6.0		3193		
2 hour Preg		1026		0.19		1.44		552	30
4 hour Preg		1027		0.18		1.50		662	30
6 hour Preg		1026		0.17		1.54		625	30
8 hour Preg		1030		0.17		1.55		619	
8 hour Dry Residue	695.4		0.47		<7		2393		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 39

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of FT22 tails (Endo/Exo varability composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	49.5	34.1	13.0
Assayed Head, g/mt			
Calculated Head, g/mt	0.69	16.7	1386
Final Tail Assay, g/mt	0.35	11.0	1205

ATS Consumption	52.0	kg ATS/metric ton ore
Ammonium Hyroxide Added	46.4	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	50.8	29.3	14.8	44.9
4	52.4	31.9	10.2	46.3
6	49.4	33.2	16.1	47.6
8	49.5	34.1	13.0	52.0

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	1748	1027	84.0	32.5	0.35			7.5	10.0	1.02
2	1748	1019				51.6		9.9		1.03
4	1749	1022				50.5		9.9		1.03
6	1751	1026				49.4		9.8		1.02
8	1748	1023				46.5		9.8		1.02
Total			84.0	32.5						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.69		16.7		1386		
2 hour Preg		1019		0.24		3.29		260	30
4 hour Preg		1022		0.24		3.47		213	30
6 hour Preg		1026		0.22		3.50		264	30
8 hour Preg		1023		0.21		3.51		232	
8 hour Dry Residue	685.6		0.35		11		1205		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 40

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of FT23 tails (Endo/Exo varability composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	77.2	28.4	7.1
Assayed Head, g/mt			
Calculated Head, g/mt	0.35	4.9	2783
Final Tail Assay, g/mt	0.08	<7	2586

ATS Consumption	54.5	kg ATS/metric ton ore
Ammonium Hyroxide Added	47.4	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	77.2	26.2	5.5	39.2
4	79.4	27.2	6.5	46.1
6	78.3	27.8	6.0	50.7
8	77.2	28.4	7.1	54.5

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	1751	1030	84.0	33.2	0.35			7.1	9.9	1.02
2	1751	1025				55.2		9.9		1.03
4	1750	1025				50.5		9.9		1.03
6	1751	1024				47.4		9.9		1.03
8	1753	1026				44.7		9.8		1.03
Total			84.0	33.2						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.35		4.9		2783		
2 hour Preg		1025		0.18		0.86		224	30
4 hour Preg		1025		0.18		0.87		241	30
6 hour Preg		1024		0.17		0.86		227	30
8 hour Preg		1026		0.17		0.86		244	
8 hour Dry Residue	688.8		0.08		<7		2586		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 41

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of FT24 tails (Endo/Exo varability composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	78.1	15.2	21.9
Assayed Head, g/mt			
Calculated Head, g/mt	0.41	42.4	4096
Final Tail Assay, g/mt	0.09	36.0	3199

ATS Consumption	45.9	kg ATS/metric ton ore
Ammonium Hyroxide Added	48.6	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	78.9	13.9	14.1	44.4
4	75.3	14.3	19.1	46.1
6	81.6	14.8	15.6	46.0
8	78.1	15.2	21.9	45.9

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	1752	1031	84.0	34.0	0.35			7.1	10.0	1.02
2	1752	1025				51.6		9.9		1.03
4	1750	1025				50.5		9.9		1.03
6	1750	1026				50.5		9.9		1.02
8	1751	1028				50.5		9.8		1.02
Total			84.0	34.0						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.41		42.4		4096		
2 hour Preg		1025		0.22		3.90		504	30
4 hour Preg		1025		0.20		3.92		631	30
6 hour Preg		1026		0.21		3.95		519	30
8 hour Preg		1028		0.19		3.91		678	
8 hour Dry Residue	680.4		0.09		36		3199		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 42

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of FT25 tails (Endo/Exo varability composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	65.8	20.7	17.5
Assayed Head, g/mt			
Calculated Head, g/mt	0.47	11.7	4153
Final Tail Assay, g/mt	0.16	9.3	3425

ATS Consumption	44.6	kg ATS/metric ton ore
Ammonium Hyroxide Added	50.3	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	66.6	17.1	14.5	39.3
4	65.4	18.8	13.6	42.9
6	65.9	19.7	9.1	44.6
8	65.8	20.7	17.5	44.6

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	1752	1031	84.0	35.2	0.35			7.5	10.0	1.02
2	1751	1024				55.2		9.9		1.03
4	1752	1024				52.7		9.9		1.03
6	1751	1023				51.6		9.9		1.03
8	1751	1023				51.6		9.9		1.03
Total			84.0	35.2						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.47		11.7		4153		
2 hour Preg		1024		0.21		1.34		525	30
4 hour Preg		1024		0.20		1.44		488	30
6 hour Preg		1023		0.20		1.47		352	30
8 hour Preg		1023		0.19		1.50		581	
8 hour Dry Residue	687.3		0.16		9.3		3425		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 43

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of FT26 tails (Endo/Exo varability composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	74.1	27.9	14.7
Assayed Head, g/mt			
Calculated Head, g/mt	0.39	4.9	2289
Final Tail Assay, g/mt	0.10	<7	1952

ATS Consumption	50.5	kg ATS/metric ton ore
Ammonium Hyroxide Added	46.4	kg NH_4OH/metric ton ore

	Extraction, % (1)			ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	74.9	24.9	11.6	41.0
4	74.8	26.3	15.5	46.0
6	76.3	27.2	14.8	48.9
8	74.1	27.9	14.7	50.5

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	1751	1031	84.0	32.5	0.35			7.5	10.0	1.02
2	1751	1026				53.9		9.9		1.03
4	1751	1026				50.5		9.8		1.02
6	1752	1028				48.4		9.8		1.02
8	1750	1026				47.4		9.9		1.02
Total			84.0	32.5						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.39		4.9		2289		
2 hour Preg		1026		0.20		0.82		301	30
4 hour Preg		1026		0.19		0.84		356	30
6 hour Preg		1028		0.19		0.84		338	30
8 hour Preg		1026		0.18		0.84		330	
8 hour Dry Residue	692.0		0.10		<7		1952		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 44

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of FT27 tails (High Grade Mag composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	34.6	18.4	20.7
Assayed Head, g/mt			
Calculated Head, g/mt	0.75	4.3	1226
Final Tail Assay, g/mt	0.49	<7	972

ATS Consumption	39.0	kg ATS/metric ton ore
Ammonium Hyroxide Added	60.0	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	36.1	16.3	18.9	35.2
4	35.9	17.2	18.5	37.3
6	34.1	17.8	21.3	37.1
8	34.6	18.4	20.7	39.0

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	Reagents Added, g NH_4OH	Reagents Added, g $CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	pH Adjust	PLS SG
0	1754	1034	84.0	42.0	0.35			6.9	10.1	1.02
2	1754	1027				57.8		9.9		1.03
4	1750	1025				56.5		9.9		1.03
6	1751	1027				56.5		9.9		1.02
8	1751	1028				55.2		9.9		1.02
Total			84.0	42.0						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.75		4.3		1226		
2 hour Preg		1027		0.18		0.48		279	30
4 hour Preg		1025		0.18		0.49		271	30
6 hour Preg		1027		0.16		0.49		290	30
8 hour Preg		1028		0.16		0.50		280	
8 hour Dry Residue	698.2		0.49		<7		972		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 46

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of LCT1 tails (Endo/Exo master composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	72.7	28.3	33.6
Assayed Head, g/mt			
Calculated Head, g/mt	0.37	22.3	3093
Final Tail Assay, g/mt	0.10	16.0	2055

ATS Consumption	54.3	kg ATS/metric ton ore
Ammonium Hyroxide Added	52.0	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	74.3	23.3	26.4	40.9
4	75.3	25.7	24.3	50.4
6	75.4	27.3	27.5	51.7
8	72.7	28.3	33.6	54.3

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	NH_4OH	$CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	Adjust	PLS SG
0	1753	1032	84.0	36.4	0.35			8.0	9.9	1.02
2	1753	1028				53.9		9.9		1.03
4	1752	1027				47.4		9.9		1.02
6	1751	1028				46.5		9.9		1.02
8	1751	1029				44.7		9.8		1.02
Total			84.0	36.4						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.37		22.3		3093		
2 hour Preg		1028		0.18		3.50		670	30
4 hour Preg		1027		0.18		3.76		612	30
6 hour Preg		1028		0.18		3.88		664	30
8 hour Preg		1029		0.16		3.92		773	
8 hour Dry Residue	691.6		0.10		16		2055		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 48

RDi Project: 23009 Empire
Date: 13-Sep-23

Purpose: To examine the extraction of gold, silver, and copper from flotatoin tails with ammonium thiosulfate.

Sample: Approximately 700 grams of LCT2 tails (Mag master composite).

Procedure: The material was transferred to a bottle and was adjusted to 50% solids. The pH of the slurry was adjusted to ~9.9 with ammonium hydroxide and ammonium thiosulfate (ATS) was added to a calculated level of 80 g/l. At 2, 4 and 6 hours, the pH and free ATS were determined. A sample of solution was removed and assayed for gold, silver, and copper. The pH was adjusted to 9.9 with ammonium thiosulfate if needed. After 8 hours, the solution was measured to determine pH, free ATS, and gold, silver and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold, silver, and copper contents.

Conditions:

Particle Size	Leach Time	ATS Concentration	% Solids
P_{80} 100 mesh	8 hours	80.0 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag	Cu
Extraction, % (1)	70.8	22.6	13.3
Assayed Head, g/mt			
Calculated Head, g/mt	0.41	25.8	7604
Final Tail Assay, g/mt	0.12	20.0	6592

ATS Consumption	62.0	kg ATS/metric ton ore
Ammonium Hyroxide Added	51.4	kg NH_4OH/metric ton ore

Extraction, % (1)				ATS Consumed
Hr.	Au	Au	Cu	kg/mt
2	69.4	19.3	12.8	41.3
4	72.0	20.8	13.5	49.2
6	71.1	22.1	13.7	54.2
8	70.8	22.6	13.3	62.0

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g ATS	Reagents Added, g NH_4OH	Reagents Added, g $CuSO_4*5H_2O$	Residual Reagents ATS g/L		pH Initial	pH Adjust	PLS SG
0	1750	1029	84.0	36.0	0.35			7.2	10.0	1.02
2	1750	1022				53.9		9.9		1.03
4	1750	1024				48.4		9.9		1.03
6	1753	1030				44.7		9.8		1.02
8	1751	1028				39.5		9.8		1.02
Total			84.0	36.0						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L	Cu g/mt	Cu mg/L	Volume Thief
Feed (analyzed)	700								
Feed (computed)			0.41		25.8		7604		
2 hour Preg		1022		0.19		3.37		781	30
4 hour Preg		1024		0.19		3.54		797	30
6 hour Preg		1030		0.19		3.63		781	30
8 hour Preg		1028		0.18		3.62		744	
8 hour Dry Residue	691.5		0.12		20		6592		

(1) Based on calculated head assays.

APPENDIX F: ACID LEACH TEST RESULTS

Bottle Roll Leaching Test 15

RDi Project: 23009 Empire
Date: 1-May-23

Purpose: To examine the extraction of copper from flotation tails at various particle sizes with sulfuric acid.

Sample: Approximately 835 grams of FT9 Tail (Endo/Exo Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 65 mesh	24 hours	13.72 g/L	38% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	78.9
Assayed Head, g/mt	
Calculated Head, g/mt	2799
Final Tail Assay, g/mt	590

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	6.2	22.049
4	67.5	41.399
8	72.4	53.470
24	78.9	71.410

Acid Consumption 71.410 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial	pH Adjust
0	2219	1385	19.0		8.0	0.4
2	2209	1375	25.3	0.0	4.8	-0.7
4	2203	1369	17.3	6.3	-0.6	
8	2184	1350	10.8	11.5	-1.0	
24	2159	1325		8.2	-0.7	
Total			72.4			

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	835				
Feed (computed)			2799		
2 hour Preg		1375		106	30
4 hour Preg		1369		1149	30
8 hour Preg		1350		1223	30
24 hour Preg		1325		1334	
24 hour Dry Residue	833.8		590		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 16

RDi Project: 23009 Empire
Date: 1-May-23

Purpose: To examine the extraction of copper from flotation tails at various particle sizes with sulfuric acid.

Sample: Approximately 855 grams of FT10 Tail (Endo/Exo Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	24 hours	14.48 g/L	39% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	80.5
Assayed Head, g/mt	
Calculated Head, g/mt	2526
Final Tail Assay, g/mt	493

Extraction, % (1) Hr.	Cu	H_2SO_4 Consumed kg/mt
2	4.7	21.533
4	70.3	42.558
8	77.3	55.183
24	80.5	72.925

Acid Consumption 72.925 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	2167	1312	19.0		8.0	#	0.8
2	2165	1311	26.0	0.0	4.8		-0.8
4	2161	1307	18.6	5.7	-0.9		
8	2162	1308	11.2	11.4	-1.0		
24	2161	1307		8.0	-0.7		
Total			74.8				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	855				
Feed (computed)			2526		
2 hour Preg		1311		77	30
4 hour Preg		1307		1159	30
8 hour Preg		1308		1247	30
24 hour Preg		1307		1272	
24 hour Dry Residue	854.4		493		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 17

RDi Project: 23009 Empire
Date: 1-May-23

Purpose: To examine the extraction of copper from flotation tails at various particle sizes with sulfuric acid.

Sample: Approximately 850 grams of FT11 Tail (Endo/Exo Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 150 mesh	24 hours	14.41 g/L	39% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	80.7
Assayed Head, g/mt	
Calculated Head, g/mt	2563
Final Tail Assay, g/mt	495

Extraction, % (1) Hr.	Cu	H_2SO_4 Consumed kg/mt
2	2.9	21.660
4	60.9	43.798
8	80.4	57.322
24	80.7	75.208

Acid Consumption 75.208 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	2169	1319	19.0		7.9	#	1.1
2	2168	1317	26.0	0.0	5.0		-1.0
4	2167	1316	19.5	5.0	-0.5		
8	2167	1316	12.0	10.8	-1.0		
24	2167	1316		8.0	-0.7		
Total			76.5				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	850				
Feed (computed)			2563		
2 hour Preg		1317		48	30
4 hour Preg		1316		1008	30
8 hour Preg		1316		1307	30
24 hour Preg		1316		1283	
24 hour Dry Residue	850.6		495		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 18

RDi Project: 23009 Empire
Date: 1-May-23

Purpose: To examine the extraction of copper from flotation tails at various particle sizes with sulfuric acid.

Sample: Approximately 850 grams of FT12 Tail (Endo/Exo Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 200 mesh	24 hours	14.55 g/L	39% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	82.3
Assayed Head, g/mt	
Calculated Head, g/mt	2761
Final Tail Assay, g/mt	490

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	1.8	21.660
4	57.4	45.703
8	72.6	61.113
24	82.3	79.594

Acid Consumption 79.594 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial	pH Adjust
0	2157	1306	19.0		7.9	0.7
2	2155	1304	25.8	0.0	5.4	-0.9
4	2159	1308	21.2	3.6	-0.5	
8	2155	1304	13.6	9.5	-1.0	
24	2155	1304		7.5	-0.7	
Total			79.6			

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	850				
Feed (computed)			2761		
2 hour Preg		1304		33.2	30
4 hour Preg		1308		1029	30
8 hour Preg		1304		1283	30
24 hour Preg		1304		1427	
24 hour Dry Residue	850.4		490		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 19

RDi Project: 23009 Empire
Date: 3-May-23

Purpose: To examine the extraction of copper from flotation tails at various particle sizes with sulfuric acid.

Sample: Approximately 725 grams of FT16 Tail (Mag Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 65 mesh	24 hours	19.66 g/L	39% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	55.5
Assayed Head, g/mt	
Calculated Head, g/mt	5793
Final Tail Assay, g/mt	2576

Extraction, % (1) Hr.	Cu	H_2SO_4 Consumed kg/mt
2	0.0	21.104
4	0.0	25.874
8	0.0	32.138
24	55.5	38.983

Acid Consumption 38.983 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial	pH Adjust
0	2109	1287	25.3		6.7	-0.8
2	2109	1286	18.1	5.7	-0.5	
4	2110	1287	4.3	16.6	-1.4	
8	2109	1287	5.3	15.8	-1.5	
24	2109	1286		15.4	-1.4	
Total			53.0			

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	825				
Feed (computed)			5793		
2 hour Preg		1286			30
4 hour Preg		1287			30
8 hour Preg		1287			30
24 hour Preg		1286		2056	
24 hour Dry Residue	822.1		2576		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 20

RDi Project: 23009 Empire
Date: 3-May-23

Purpose: To examine the extraction of copper from flotation tails at various particle sizes with sulfuric acid.

Sample: Approximately 725 grams of FT16 Tail (Mag Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	24 hours	19.68 g/L	39% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	58.9
Assayed Head, g/mt	
Calculated Head, g/mt	6487
Final Tail Assay, g/mt	2664

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	50.4	20.927
4	51.9	25.489
8	51.6	31.103
24	58.9	38.871

Acid Consumption 38.871 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial	pH Adjust
0	1991	1215	23.9		6.9	-0.9
2	1990	1214	16.9	5.9	-0.6	
4	1991	1215	3.9	16.8	-1.5	
8	1992	1216	4.4	16.3	-1.5	
24	1993	1217		14.8	-1.3	
Total			49.1			

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	775				
Feed (computed)			6487		
2 hour Preg		1214		2092	30
4 hour Preg		1215		2100	30
8 hour Preg		1216		2033	30
24 hour Preg		1217		2285	
24 hour Dry Residue	776.2		2664		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 21

RDi Project: 23009 Empire
Date: 3-May-23

Purpose: To examine the extraction of copper from flotation tails at various particle sizes with sulfuric acid.

Sample: Approximately 725 grams of FT16 Tail (Mag Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 150 mesh	24 hours	20.46 g/L	41% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	58.0
Assayed Head, g/mt	
Calculated Head, g/mt	6441
Final Tail Assay, g/mt	2703

Extraction, % (1) Hr.	Cu	H_2SO_4 Consumed kg/mt
2	43.8	20.383
4	43.4	25.330
8	60.2	30.841
24	58.0	38.239

Acid Consumption 38.239 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial	pH Adjust
0	1856	1090	22.3		7.3	-0.9
2	1855	1089	15.9	5.7	-0.6	
4	1856	1090	3.7	16.7	-1.4	
8	1856	1090	4.3	16.1	-1.5	
24	1855	1089		14.7	-1.3	
Total			46.2			

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	765				
Feed (computed)			6441		
2 hour Preg		1089		1982	30
4 hour Preg		1090		1910	30
8 hour Preg		1090		2618	30
24 hour Preg		1089		2449	
24 hour Dry Residue	765.8		2703		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 22

RDi Project: 23009 Empire
Date: 3-May-23

Purpose: To examine the extraction of copper from flotation tails at various particle sizes with sulfuric acid.

Sample: Approximately 725 grams of FT16 Tail (Mag Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 200 mesh	24 hours	18.94 g/L	37% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	63.8
Assayed Head, g/mt	
Calculated Head, g/mt	7660
Final Tail Assay, g/mt	2775

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	54.5	23.718
4	56.5	29.597
8	56.1	36.738
24	63.8	45.170

Acid Consumption 45.170 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1971	1246	23.6		7.2	#	-0.7
2	1970	1246	18.1	4.7	-0.6		
4	1970	1246	5.1	15.7	-1.4		
8	1971	1246	5.3	15.5	-1.5		
24	1970	1245		14.7	-1.4		
Total			52.1				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	725				
Feed (computed)			7660		
2 hour Preg		1246		2427	30
4 hour Preg		1246		2461	30
8 hour Preg		1246		2382	30
24 hour Preg		1245		2668	
24 hour Dry Residue	724.8		2775		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 25

RDi Project: 23009 Empire
Date: 3-May-23

Purpose: To examine the extraction of copper from ATS leach residue with sulfuric acid.

Sample: Approximately 965 grams of BR23 Residue (Endo/Exo Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	24 hours	26.06 g/L	39% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	72.0
Assayed Head, g/mt	
Calculated Head, g/mt	2757
Final Tail Assay, g/mt	773

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	48.5	39.396
4	72.0	53.255
8	75.8	65.567
24	72.0	82.527

Acid Consumption 82.527 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	2500	1535	40.0		9.0	#	-1.0
2	2497	1533	29.3	0.5	0.2		
4	2499	1534	14.1	10.6	-1.5		
8	2499	1534	12.3	11.8	-1.7		
24	2500	1536		8.8	-0.9		
Total			95.7				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	965				
Feed (computed)			2757		
2 hour Preg		1533		842	30
4 hour Preg		1534		1232	30
8 hour Preg		1534		1274	30
24 hour Preg		1536		1181	
24 hour Dry Residue	964.6		773		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 26

RDi Project: 23009 Empire
Date: 16-May-23

Purpose: To examine the extraction of copper from ATS leach residue with sulfuric acid.

Sample: Approximately 965 grams of BR24 Residue (Mag Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	24 hours	19.54 g/L	39% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	56.7
Assayed Head, g/mt	
Calculated Head, g/mt	6187
Final Tail Assay, g/mt	2678

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	49.6	23.959
4	51.5	31.219
8	55.6	35.945
24	56.7	42.319

Acid Consumption 42.319 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	2500	1536	30.0		9.2	#	-1.4
2	2499	1535	24.0	4.0	-0.8		
4	2502	1538	7.6	14.9	-1.6		
8	2500	1536	4.8	16.8	-1.8		
24	2499	1535		15.8	-1.1		
Total			66.4				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	965				
Feed (computed)			6187		
2 hour Preg		1535		1929	30
4 hour Preg		1538		1962	30
8 hour Preg		1536		2085	30
24 hour Preg		1535		2087	
24 hour Dry Residue	964.1		2678		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 28

RDi Project: 23009 Empire
Date: 17-Jul-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 1000 grams of FT17 Residue (Endo/Exo Master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 8 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 24 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	24 hours	19.72 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	82.2
Assayed Head, g/mt	
Calculated Head, g/mt	3080
Final Tail Assay, g/mt	549

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	44.3	28.629
4	74.1	45.815
8	80.8	57.303
24	82.2	74.856

Acid Consumption 74.856 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	2501	1521	30.0		7.7	#	1.5
2	2496	1517	29.6	0.3	2.6		1.3
4	2500	1521	17.8	8.1	1.5		
8	2501	1522	12.0	12.0	1.5		
24	2498	1519		8.0	1.6		
Total			89.4				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	1000				
Feed (computed)			3080		
2 hour Preg		1517		880	30
4 hour Preg		1521		1452	30
8 hour Preg		1522		1555	30
24 hour Preg		1519		1555	
24 hour Dry Residue	979.3		549		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 29

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of FT20 Tail (Endo/Exo varability composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.55 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	75.6
Assayed Head, g/mt	
Calculated Head, g/mt	4683
Final Tail Assay, g/mt	1142

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	78.2	20.299
4	63.8	27.199
6	70.3	33.107
8	75.6	37.964

Acid Consumption 37.964 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1750	1074	21.0		7.5	#	1.1
2	1750	1074	14.8	5.9	1.3		
4	1760	1084	5.4	14.9	1.1		
6	1750	1074	4.1	16.1	0.8		
8	1753	1077		16.6	0.8		
Total			45.3				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			4683		
2 hour Preg		1074		2307	30
4 hour Preg		1084		1800	30
6 hour Preg		1074		1960	30
8 hour Preg		1077		2055	
8 hour Dry Residue	676.3		1142		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 30

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of FT21 Tail (Endo/Exo varability composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.78 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	76.5
Assayed Head, g/mt	
Calculated Head, g/mt	3606
Final Tail Assay, g/mt	849

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	76.0	13.200
4	90.3	16.302
6	77.0	18.943
8	76.5	20.973

Acid Consumption 20.973 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1750	1062	21.0		7.3	#	1.1
2	1750	1062	9.7	10.8	1.2		
4	1752	1063	2.3	17.8	1.0		
6	1750	1061	1.9	18.2	0.8		
8	1751	1062		18.6	0.7		
Total			34.9				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			3606		
2 hour Preg		1062		1777	30
4 hour Preg		1063		2059	30
6 hour Preg		1061		1694	30
8 hour Preg		1062		1632	
8 hour Dry Residue	688.8		849		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 31

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of FT22 Tail (Endo/Exo varability composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.67 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	39.3
Assayed Head, g/mt	
Calculated Head, g/mt	1516
Final Tail Assay, g/mt	920

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	24.1	23.760
4	30.6	33.563
6	43.1	41.398
8	39.3	47.479

Acid Consumption 47.479 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1751	1068	21.0		7.8	#	1.1
2	1749	1066	17.2	3.6	1.3		
4	1765	1082	7.5	12.9	1.0		
6	1750	1067	5.5	14.8	0.8		
8	1753	1070		15.8	0.8		
Total			51.2				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			1516		
2 hour Preg		1066		234	30
4 hour Preg		1082		287	30
6 hour Preg		1067		404	30
8 hour Preg		1070		355	
8 hour Dry Residue	683.2		920		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 32

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of FT23 Tail (Endo/Exo varability composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.67 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	43.1
Assayed Head, g/mt	
Calculated Head, g/mt	3203
Final Tail Assay, g/mt	1821

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	26.2	14.888
4	36.0	19.028
6	39.8	22.622
8	43.1	25.138

Acid Consumption 25.138 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1750	1067	21.0		7.4	#	1.1
2	1750	1067	10.9	9.6	1.1		
4	1750	1068	3.2	17.0	1.0		
6	1754	1072	2.6	17.5	0.8		
8	1750	1068		18.3	0.7		
Total			37.7				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			3203		
2 hour Preg		1067		536	30
4 hour Preg		1068		721	30
6 hour Preg		1072		777	30
8 hour Preg		1068		826	
8 hour Dry Residue	682.5		1821		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 33

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of FT24 Tail (Endo/Exo varability composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.28 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	64.4
Assayed Head, g/mt	
Calculated Head, g/mt	4663
Final Tail Assay, g/mt	1658

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	48.2	27.111
4	60.1	38.389
6	67.4	45.375
8	64.4	51.831

Acid Consumption 51.831 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1752	1089	21.0		7.4	#	1.1
2	1752	1089	19.6	1.3	1.6		
4	1753	1091	8.6	11.8	1.1		
6	1750	1088	5.1	15.1	0.8		
8	1750	1088		15.5	0.7		
Total			54.3				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			4663		
2 hour Preg		1089		1368	30
4 hour Preg		1091		1667	30
6 hour Preg		1088		1833	30
8 hour Preg		1088		1697	
8 hour Dry Residue	662.7		1658		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 34

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of FT25 Tail (Endo/Exo varability composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.66 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	74.9
Assayed Head, g/mt	
Calculated Head, g/mt	5413
Final Tail Assay, g/mt	1358

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	78.6	23.016
4	76.2	35.140
6	79.2	39.556
8	74.9	42.810

Acid Consumption 42.810 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1750	1068	21.0		7.6	#	1.0
2	1749	1067	15.6	4.1	1.3		
4	1750	1068	14.2	10.5	1.0		
6	1751	1069		20.8	0.7		
8	1751	1069		18.6	0.7		
Total			50.8				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			5413		
2 hour Preg		1067		2722	30
4 hour Preg		1068		2557	30
6 hour Preg		1069		2589	30
8 hour Preg		1069		2368	
8 hour Dry Residue	682.2		1358		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 35

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of FT26 Tail (Endo/Exo varability composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.66 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	62.4
Assayed Head, g/mt	
Calculated Head, g/mt	2951
Final Tail Assay, g/mt	1109

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	49.8	15.615
4	64.8	20.916
6	65.0	24.057
8	62.4	26.983

Acid Consumption 26.983 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1752	1068	21.0		7.6	#	1.0
2	1752	1068	11.4	9.1	1.1		
4	1751	1067	4.0	16.2	1.0		
6	1752	1069	2.3	17.8	0.8		
8	1751	1067		18.0	0.7		
Total			38.7				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			2951		
2 hour Preg		1068		940	30
4 hour Preg		1067		1198	30
6 hour Preg		1069		1168	30
8 hour Preg		1067		1087	
8 hour Dry Residue	683.7		1109		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 36

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of FT27 Tail (High Grade Mag composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.85 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	45.2
Assayed Head, g/mt	
Calculated Head, g/mt	1229
Final Tail Assay, g/mt	673

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	33.2	8.133
4	33.8	11.113
6	41.3	12.864
8	45.2	14.559

Acid Consumption 14.559 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1751	1058	21.0		7.0	#	1.0
2	1751	1058	6.0	14.3	1.0		
4	1753	1060	2.2	17.9	1.0		
6	1752	1059	1.3	18.8	0.7		
8	1750	1057		18.9	0.7		
Total			30.5				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			1229		
2 hour Preg		1058		267	30
4 hour Preg		1060		264	30
6 hour Preg		1059		317	30
8 hour Preg		1057		340	
8 hour Dry Residue	692.8		673		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 45

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of LCT1 Tail (Endo/Exo master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.53 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	84.3
Assayed Head, g/mt	
Calculated Head, g/mt	3916
Final Tail Assay, g/mt	614

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	38.8	28.922
4	74.8	46.787
6	72.1	55.455
8	84.3	61.198

Acid Consumption 61.198 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1759	1076	21.0		8.1	#	1.3
2	1754	1071	22.5	0.1	2.5		
4	1750	1066	11.4	9.1	1.1		
6	1751	1068	6.4	13.9	1.0		
8	1752	1068		16.0	0.9		
Total			61.3				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			3916		
2 hour Preg		1071		970	30
4 hour Preg		1066		1851	30
6 hour Preg		1068		1729	30
8 hour Preg		1068		1985	
8 hour Dry Residue	683.4		614		

(1) Based on calculated head assays.

Bottle Roll Leaching Test 47

RDi Project: 23009 Empire
Date: 11-Sep-23

Purpose: To examine the extraction of copper from flotation tails with sulfuric acid.

Sample: Approximately 700 grams of LCT2 Tail (Mag master composite)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. Sulfuric acid was added to a calculated level of 20.0 g/L and the pH was recorded. At 2, 4, and 6 hours, the pH and free acid were determined. A sample of solution was removed and assayed for copper. Sulfuric acid was added to return the level to 20.0 g/L and the pH was recorded. After 8 hours, the solution was measured to determine pH, free acid, and copper contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for copper assay.

Conditions:

Particle Size	Leach Time	H_2SO_4 Concentration	% Solids
P_{80} 100 mesh	8 hours	19.34 g/L	40% Solids

Summary of Results:

Parameter	Cu
Extraction, % (1)	62.3
Assayed Head, g/mt	
Calculated Head, g/mt	7885
Final Tail Assay, g/mt	2970

Extraction, % (1)		H_2SO_4 Consumed
Hr.	Cu	kg/mt
2	50.7	21.559
4	54.5	28.727
6	60.0	32.807
8	62.3	36.466

Acid Consumption 36.466 kg H_2SO_4/metric ton ore

Detailed Results:

A. Leach Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g H_2SO_4 @96.9%	Residual Reagents H_2SO_4 g/L	pH Initial		pH Adjust
0	1753	1086	21.0		7.6	#	1.2
2	1752	1085	15.8	5.0	1.3		
4	1751	1084	5.4	14.8	1.0		
6	1749	1082	3.1	17.1	1.0		
8	1763	1096		17.3	0.9		
Total			45.3				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Cu mg/kg	Cu mg/L	Volume Thief
Feed (analyzed)	700				
Feed (computed)			7885		
2 hour Preg		1085		2455	30
4 hour Preg		1084		2574	30
6 hour Preg		1082		2777	30
8 hour Preg		1096		2778	
8 hour Dry Residue	667.0		2970		

(1) Based on calculated head assays.

APPENDIX G: FLOTATION TEST RESULTS

RDi Project No.: **23009 Empire**
Flotation Test No. 1 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes.

Sample: Approximately 1 kg of GT1 Tails (Endo/Exo Master Composite).
Approximately P_{80} of 65 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	
Operations	min.	Solids %	pH-Start	pH-End	eH					
Rod Mill Grinding	0.00	50		8.7	233	0	0	0	0	
Conditioning	1.0	26	8.7	8.8	206	50	25	25	20	
Ro Flotation - 1	3.0	26	8.8	8.8	213					
Conditioning	1.0	26	8.8	8.8	177	25	25	25	10	
Ro Flotation - 2	3.0	26	8.8	8.8	195					
Conditioning	1.0	26	8.8	8.9	158	25	25	25	10	
Ro Flotation - 3	3.0	26	8.9	8.9	178					
Total Reagent Used, g/mt of feed						100	75	75	40	
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	972.2	100.0	0.147	18.140	0.426	100.0	100.0	100.0
Rougher Conc-1	6.5	0.7	6.24	505	9.32	28.4	18.6	14.6
Rougher Conc-2/3	19.2	2.0	0.89	91.9	1.53	12.0	10.0	7.1
Ro-Tail	946.5	97.4	0.09	13.3	0.34	59.6	71.4	78.3

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	25.7	2.6	2.24	196.4	3.50	40.4	28.6	21.7

RDi Project No.: **23009 Empire**
Flotation Test No. 2 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes.

Sample: Approximately 1 kg of GT2 Tails (Endo/Exo Master Composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	
Operations	min.	Solids %	pH-Start	pH-End	eH					
Rod Mill Grinding	0.00	50		8.7	243	0	0	0	0	
Conditioning	1.0	26	8.7	8.8	187	50	25	25	20	
Ro Flotation - 1	3.0	26	8.8	8.8	201					
Conditioning	1.0	26	8.8	8.8	161	25	25	25	10	
Ro Flotation - 2	3.0	26	8.8	8.8	182					
Conditioning	1.0	26	8.8	8.8	129	25	25	25	10	
Ro Flotation - 3	3.0	26	8.8	8.8	167					
Total Reagent Used, g/mt of feed						100	75	75	40	
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	986.5	100.0	0.140	17.299	0.428	100.0	100.0	100.0
Rougher Conc-1	14.1	1.4	3.20	285.0	4.42	32.6	23.5	14.7
Rougher Conc-2/3	25.5	2.6	0.69	84.6	1.37	12.7	12.6	8.3
Ro-Tail	946.9	96.0	0.08	11.5	0.34	54.7	63.8	77.0

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	39.6	4.0	1.58	156.0	2.46	45.3	36.2	23.0

RDi Project No.: **23009 Empire**
Flotation Test No. 3 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes.

Sample: Approximately 1 kg of GT3 Tails (Endo/Exo Master Composite).
Approximately P_{80} of 150 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	
Operations	min.	Solids %	pH-Start	pH-End	eH					
Rod Mill Grinding	0.00	50		8.8	152	0	0	0	0	
Conditioning	1.0	26	8.8	8.8	161	50	25	25	20	
Ro Flotation - 1	3.0	26	8.8	8.8	175					
Conditioning	1.0	26	8.8	8.8	156	25	25	25	10	
Ro Flotation - 2	3.0	26	8.8	8.8	170					
Conditioning	1.0	26	8.8	8.9	145	25	25	25	10	
Ro Flotation - 3	3.0	26	8.9	8.9	163					
Total Reagent Used, g/mt of feed						100	75	75	40	
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	994.7	100.0	0.123	21.412	0.438	100.0	100.0	100.0
Rougher Conc-1	11.5	1.2	3.77	420	7.24	35.4	22.7	19.1
Rougher Conc-2/3	24.8	2.5	0.49	80.5	1.02	9.9	9.4	5.8
Ro-Tail	958.4	96.4	0.07	15.1	0.34	54.7	67.9	75.1

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	36.3	3.6	1.53	188.1	2.99	45.3	32.1	24.9

RDi Project No.: **23009 Empire**
Flotation Test No. 4 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes.

Sample: Approximately 1 kg of GT4 Tails (Endo/Exo Master Composite).
Approximately P_{80} of 200 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	
Operations	min.	Solids %	pH-Start	pH-End	eH					
Rod Mill Grinding	0.00	50		8.6	191	0	0	0	0	
Conditioning	1.0	26	8.6	8.7	169	50	25	25	20	
Ro Flotation - 1	3.0	26	8.7	8.7	168					
Conditioning	1.0	26	8.7	8.7	154	25	25	25	10	
Ro Flotation - 2	3.0	26	8.7	8.8	159					
Conditioning	1.0	26	8.8	8.8	145	25	25	25	10	
Ro Flotation - 3	3.0	26	8.8	8.8	147					
Total Reagent Used, g/mt of feed						100	75	75	40	
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	995.0	100.0	0.135	19.454	0.414	100.0	100.0	100.0
Rougher Conc-1	27.4	2.8	1.67	115.0	1.92	34.0	16.3	12.8
Rougher Conc-2/3	30.2	3.0	0.46	67.9	1.15	10.3	10.6	8.4
Ro-Tail	937.4	94.2	0.08	15.1	0.35	55.7	73.1	78.8

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	57.6	5.8	1.04	90.3	1.51	44.3	26.9	21.2

RDi Project No.: **23009 Empire**
Flotation Test No. 5 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes.

Sample: Approximately 1 kg of GT5 Tails (Mag Master Composite).
Approximately P_{80} of 65 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	
Operations	min.	Solids %	pH-Start	pH-End	eH					
Rod Mill Grinding	0.00	50		8.3	273	0	0	0	0	
Conditioning	1.0	26	8.3	8.3	196	50	25	25	20	
Ro Flotation - 1	3.0	26	8.3	8.4	178					
Conditioning	1.0	26	8.4	8.5	147	25	25	25	10	
Ro Flotation - 2	3.0	26	8.5	8.5	154					
Conditioning	1.0	26	8.5	8.5	133	25	25	25	10	
Ro Flotation - 3	3.0	26	8.5	8.6	146					
Total Reagent Used, g/mt of feed						100	75	75	40	
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	976.3	100.0	0.139	16.388	0.647	100.0	100.0	100.0
Rougher Conc-1	70.5	7.2	0.50	19.5	0.77	25.9	8.6	8.5
Rougher Conc-2/3	29.4	3.0	0.45	20.5	0.87	9.7	3.8	4.0
Ro-Tail	876.4	89.8	0.10	16.0	0.63	64.4	87.6	87.4

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	99.9	10.2	0.49	19.8	0.80	35.6	12.4	12.6

RDi Project No.: **23009 Empire**
Flotation Test No. 6 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes.

Sample: Approximately 1 kg of GT6 Tails (Mag Master Composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	
Operations	min.	Solids %	pH-Start	pH-End	eH					
Rod Mill Grinding	0.00	50		8.2	291	0	0	0	0	
Conditioning	1.0	26	8.2	8.3	216	50	25	25	20	
Ro Flotation - 1	3.0	26	8.3	8.3	190					
Conditioning	1.0	26	8.3	8.3	167	25	25	25	10	
Ro Flotation - 2	3.0	26	8.3	8.5	144					
Conditioning	1.0	26	8.5	8.4	134	25	25	25	10	
Ro Flotation - 3	3.0	26	8.4	8.5	150					
Total Reagent Used, g/mt of feed						100	75	75	40	
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	972.1	100.0	0.137	10.596	0.679	100.0	100.0	100.0
Rougher Conc-1	43.7	4.5	0.61	29.4	0.80	20.0	12.5	5.3
Rougher Conc-2/3	30.6	3.1	0.55	33.2	0.85	12.6	9.9	3.9
Ro-Tail	897.8	92.4	0.10	8.9	0.67	67.4	77.7	90.8

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	74.3	7.6	0.59	31.0	0.82	32.6	22.3	9.2

RDi Project No.: **23009 Empire**
Flotation Test No. 7 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes.

Sample: Approximately 1 kg of GT7 Tails (Mag Master Composite).
Approximately P_{80} of 150 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	
Operations	min.	Solids %	pH-Start	pH-End	eH					
Rod Mill Grinding	0.00	50		8.3	223	0	0	0	0	
Conditioning	1.0	26	8.3	8.3	212	50	25	25	20	
Ro Flotation - 1	3.0	26	8.3	8.4	205					
Conditioning	1.0	26	8.4	8.4	178	25	25	25	10	
Ro Flotation - 2	3.0	26	8.4	8.5	175					
Conditioning	1.0	26	8.5	8.5	165	25	25	25	10	
Ro Flotation - 3	3.0	26	8.5	8.5	167					
Total Reagent Used, g/mt of feed						100	75	75	40	
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	985.7	100.0	0.130	15.150	0.686	100.0	100.0	100.0
Rougher Conc-1	81.9	8.3	0.46	24.7	0.75	29.3	13.5	9.0
Rougher Conc-2/3	27.2	2.8	0.44	46.0	0.85	9.3	8.4	3.4
Ro-Tail	876.6	88.9	0.09	13.3	0.68	61.4	78.1	87.6

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	109.1	11.1	0.46	30.0	0.77	38.6	21.9	12.4

RDi Project No.: **23009 Empire**
Flotation Test No. 8 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes.

Sample: Approximately 1 kg of GT8 Tails (Mag Master Composite).
Approximately P_{80} of 200 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	
Operations	min.	Solids %	pH-Start	pH-End	eH					
Rod Mill Grinding	0.00	50		8.2	189	0	0	0	0	
Conditioning	1.0	26	8.2	8.2	176	50	25	25	20	
Ro Flotation - 1	3.0	26	8.2	8.3	183					
Conditioning	1.0	26	8.3	8.3	187	25	25	25	10	
Ro Flotation - 2	3.0	26	8.3	8.3	179					
Conditioning	1.0	26	8.3	8.4	183	25	25	25	10	
Ro Flotation - 3	3.0	26	8.4	8.5	182					
Total Reagent Used, g/mt of feed						100	75	75	40	
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	997.4	100.0	0.134	14.807	0.719	100.0	100.0	100.0
Rougher Conc-1	79.8	8.0	0.42	22.1	0.80	25.1	11.9	9.0
Rougher Conc-2/3	64.5	6.5	0.36	31.0	0.94	17.4	13.5	8.4
Ro-Tail	853.1	85.5	0.09	12.9	0.69	57.5	74.5	82.6

.

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	144.3	14.5	0.39	26.1	0.86	42.5	25.5	17.4

RDi Project No.: **23009 Empire**
Flotation Test No. 9 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes with sulfidization.

Sample: Approximately 1 kg of GT1 Tails (Endo/Exo Master Composite).
Approximately P_{80} of 65 mesh

Procedure:

	Conditions				Reagents, g/mt of flotation feed					
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	0.00	50		8.7	242	0	0	0	0	0
Conditioning	1.0	26	8.7	10.0	-144	50	25	25	20	500
Ro Flotation - 1	3.0	26	10.0	10.0	-33					
Conditioning	1.0	26	10.0	9.9	-200	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.9	9.9	-64					
Conditioning	1.0	26	9.9	10.0	-103	25	25	25	10	63
Ro Flotation - 3	3.0	26	10.0	9.9	-64					
Total Reagent Used, g/mt of feed						100	75	75	40	625
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	976.3	100.0	0.128	16.847	0.471	100.0	100.0	100.0
Rougher Conc-1	13.9	1.4	3.09	522	12.33	34.4	44.1	37.2
Rougher Conc-2/3	21.6	2.2	0.75	125	1.78	13.0	16.4	8.3
Ro-Tail	940.8	96.4	0.07	6.9	0.27	52.7	39.5	54.4

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	35.5	3.6	1.67	280.4	5.91	47.3	60.5	45.6

RDi Project No.: **23009 Empire**
Flotation Test No. 10 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes with sulfidization.

Sample: Approximately 1 kg of GT2 Tails (Endo/Exo Master Composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	0.00	50		8.7	191	0	0	0	0	0
Conditioning	1.0	26	8.7	9.6	-142	50	25	25	20	375
Ro Flotation - 1	3.0	26	9.6	9.5	-13					
Conditioning	1.0	26	9.5	9.7	-97	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.7	9.6	-23					
Conditioning	1.0	26	9.6	9.8	-124	25	25	25	10	63
Ro Flotation - 3	3.0	26	9.8	9.6	-57					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	984.7	100.0	0.141	30.135	0.456	100.0	100.0	100.0
Rougher Conc-1	12.1	1.2	4.76	765	14.30	41.5	31.2	38.5
Rougher Conc-2/3	16.9	1.7	0.85	128	1.84	10.3	7.3	6.9
Ro-Tail	955.7	97.1	0.07	19.1	0.26	48.2	61.5	54.6

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	29.0	2.9	2.48	393.8	7.04	51.8	38.5	45.4

RDi Project No.: **23009 Empire**
Flotation Test No. 11 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes with sulfidization.

Sample: Approximately 1 kg of GT3 Tails (Endo/Exo Master Composite).
Approximately P_{80} of 150 mesh

Procedure:

	Conditions				Reagents, g/mt of flotation feed					
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	0.00	50		8.7	144	0	0	0	0	0
Conditioning	1.0	26	8.7	9.4	-106	50	25	25	20	375
Ro Flotation - 1	3.0	26	9.4	9.3	13					
Conditioning	1.0	26	9.3	9.5	-107	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.5	9.5	-38					
Conditioning	1.0	26	9.5	9.6	-128	25	25	25	10	63
Ro Flotation - 3	3.0	26	9.6	9.6	-61					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	988.8	100.0	0.139	18.410	0.481	100.0	100.0	100.0
Rougher Conc-1	12.8	1.3	3.83	749	13.55	35.8	52.7	36.4
Rougher Conc-2/3	15.8	1.6	0.71	157	3.00	8.2	13.6	10.0
Ro-Tail	960.2	97.1	0.08	6.4	0.27	56.0	33.7	53.6

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	28.6	2.9	2.11	422.0	7.73	44.0	66.3	46.4

RDi Project No.: **23009 Empire**
Flotation Test No. 12 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes with sulfidization.

Sample: Approximately 1 kg of GT4 Tails (Endo/Exo Master Composite).
Approximately P_{80} of 200 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	0.00	50		8.6	145	0	0	0	0	0
Conditioning	1.0	26	8.6	9.4	-115	50	25	25	20	375
Ro Flotation - 1	3.0	26	9.4	9.3	-13					
Conditioning	1.0	26	9.3	9.4	-167	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.4	9.3	-47					
Conditioning	1.0	26	9.3	9.5	-135	25	25	25	10	63
Ro Flotation - 3	3.0	26	9.5	9.4	-71					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	985.7	100.0	0.146	18.760	0.448	100.0	100.0	100.0
Rougher Conc-1	10.3	1.0	5.05	1004	14.48	36.1	55.9	33.7
Rougher Conc-2/3	22.5	2.3	0.71	167	2.14	11.1	20.3	10.9
Ro-Tail	952.9	96.7	0.08	4.6	0.26	52.9	23.8	55.4

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	32.8	3.3	2.07	429.8	6.01	47.1	76.2	44.6

RDi Project No.: **23009 Empire**
Flotation Test No. 13 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes with sulfidization.

Sample: Approximately 1 kg of GT5 Tails (Mag Master Composite).
Approximately P_{80} of 65 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	0.00	50		8.2	187	0	0	0	0	0
Conditioning	1.0	26	8.2	9.4	-27	50	25	25	20	500
Ro Flotation - 1	3.0	26	9.4	9.2	38					
Conditioning	1.0	26	9.2	9.5	-53	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.5	9.3	14					
Conditioning	1.0	26	9.3	9.6	-80	25	25	25	10	63
Ro Flotation - 3	3.0	26	9.6	9.4	-1					
Total Reagent Used, g/mt of feed						100	75	75	40	625
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	980.2	100.0	0.158	17.367	0.668	100.0	100.0	100.0
Rougher Conc-1	22.9	2.3	1.83	64.8	1.33	27.1	8.7	4.7
Rougher Conc-2/3	50.2	5.1	0.44	62.0	1.22	14.3	18.3	9.3
Ro-Tail	907.1	92.5	0.10	13.7	0.62	58.6	73.0	86.0

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	73.1	7.5	0.88	62.9	1.25	41.4	27.0	14.0

RDi Project No.: **23009 Empire**
Flotation Test No. 14 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes with sulfidization.

Sample: Approximately 1 kg of GT6 Tails (Mag Master Composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	0.00	50		8.2	208	0	0	0	0	0
Conditioning	1.0	25	8.2	9.5	-63	50	25	25	20	500
Ro Flotation - 1	3.0	25	9.5	9.2	18					
Conditioning	1.0	25	9.2	9.5	-76	25	25	25	10	63
Ro Flotation - 2	3.0	25	9.5	9.3	10					
Conditioning	1.0	25	9.3	9.6	-78	25	25	25	10	63
Ro Flotation - 3	3.0	25	9.6	9.5	-13					
Total Reagent Used, g/mt of feed						100	75	75	40	625
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	970.3	100.0	0.148	12.319	0.673	100.0	100.0	100.0
Rougher Conc-1	25.6	2.6	1.13	71.6	1.37	20.1	15.3	5.4
Rougher Conc-2/3	59.9	6.2	0.44	30.1	1.12	18.3	15.1	10.3
Ro-Tail	884.8	91.2	0.10	9.4	0.62	61.5	69.6	84.3

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	85.5	8.8	0.65	42.5	1.20	38.5	30.4	15.7

RDi Project No.: **23009 Empire**
Flotation Test No. 15 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes with sulfidization.

Sample: Approximately 1 kg of GT7 Tails (Mag Master Composite).
Approximately P_{80} of 150 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	0.00	50		8.2	206	0	0	0	0	0
Conditioning	1.0	26	8.2	9.3	-71	50	25	25	20	500
Ro Flotation - 1	3.0	26	9.3	9.2	10					
Conditioning	1.0	26	9.2	9.5	-94	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.5	9.4	-13					
Conditioning	1.0	26	9.4	9.5	-113	25	25	25	10	63
Ro Flotation - 3	3.0	26	9.5	9.4	-19					
Total Reagent Used, g/mt of feed						100	75	75	40	625
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	979.7	100.0	0.128	12.239	0.657	100.0	100.0	100.0
Rougher Conc-1	40.7	4.2	0.73	84.3	1.29	23.6	28.6	8.2
Rougher Conc-2/3	110.1	11.2	0.27	21.2	0.96	23.6	19.5	16.4
Ro-Tail	828.9	84.6	0.08	7.5	0.59	52.7	51.9	75.4

Observations:

Flotation in a 3.2 liter cell at 1650 RPM

Total Combined Conc	150.8	15.4	0.39	38.2	1.05	47.3	48.1	24.6

RDi Project No.: **23009 Empire**
Flotation Test No. 16 **Date:** **25-Apr-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from gravity tails at various particle sizes with sulfidization.

Sample: Approximately 1 kg of GT8 Tails (Mag Master Composite).
Approximately P_{80} of 200 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	0.00	50		8.2	190	0	0	0	0	0
Conditioning	1.0	26	8.2	9.5	-115	50	25	25	20	500
Ro Flotation - 1	3.0	26	9.5	9.3	21					
Conditioning	1.0	26	9.3	9.2	-20	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.2	9.1	39					
Conditioning	1.0	26	9.1	9.5	-91	25	25	25	10	63
Ro Flotation - 3	3.0	26	9.5	9.3	-3					
Total Reagent Used, g/mt of feed						100	75	75	40	625
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	1,000							
Feed (calculated)	986.3	100.0	0.151	12.442	0.647	100.0	100.0	100.0
Rougher Conc-1	28.1	2.8	0.90	80.3	1.43	17.0	18.4	6.3
Rougher Conc-2/3	81.6	8.3	0.55	36.9	1.06	30.1	24.5	13.6
Ro-Tail	876.6	88.9	0.09	8.0	0.58	52.9	57.1	80.1

Observations:

	Flotation in a 3.2 liter cell at 1650 RPM							
Total Combined Conc	109.7	11.1	0.64	48.0	1.16	47.1	42.9	19.9

RDi Project No.: **23009 Empire**
Flotation Test No. 17 **Date:** **17-Jul-23**

Purpose: Preliminary flotation test to recover a bulk concentrate for cleaner flotation testing.

Sample: Approximately 20 kg of Endo/Exo Master Composite.
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	22.0	50		8.1/8.5	68/87	0	0	0	0	0
Conditioning	1.0	24	8.1/8.5	8.7/8.9	-167/-113	50	25	25	4	375
Ro Flotation - 1	4.5	24	8.7/8.9	8.7/8.8	26/-15					
Conditioning	1.0	24	8.7/8.8	8.9/8.9	-176/-163	25	25	25	2	63
Ro Flotation - 2	4.5	24	8.9/8.9	8.8/8.8	-6/-20					
Conditioning	1.0	24	8.8/8.8	8.9/8.9	-141/-147	25	25	25	2	63
Ro Flotation - 3	4.5	24	8.9/8.9	8.8/8.8	7/0					
Total Reagent Used, g/mt of feed						100	75	75	8	500
Solution Concentration (% or g/drop)						5.0%	5.0%	5.0%	0.0050	25.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	20,000							
Feed (calculated)	19,887	100.0	0.15	21.0	0.345	100.0	100.0	100.0
Rougher Conc	257.1	1.3	6.05	762	10.47	51.4	46.9	39.3
Ro-Tail	19,630	98.7	0.08	11.3	0.21	48.6	53.1	60.7

Observations:

	Flotation in cubic foot cell							
Total Combined Conc	257.1	1.3	6.05	761.5	10.47	51.4	46.9	39.3

RDi Project No.: **23009 Empire**
Flotation Test No. 18 **Date:** **21-Jul-23**

Purpose: Preliminary cleaner flotation test to recover a cleaned copper concentrate.

Sample: Approximately 130 grams of FT17 concentrate (Endo/Exo Master Composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Pebble Mill Grinding	0.0	50		7.2	145	0	0	0	0	0
Conditioning	1.0	10	7.2	8.0	-111	77	154	154	77	1538
Cleaner 1 Flotation	6.0	10	8.0	8.1	-2					
Conditioning	1.0	3	8.1	8.2	24	0	77	77	77	0
Cleaner 2 Flotation	2.0	3	8.2	8.0	62					
Total Reagent Used, g/mt of feed						77	231	231	154	1538
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	130							
Feed (calculated)	127.6	100.0	6.85	767	10.51	100.0	100.0	100.0
Cleaner 2 Conc	29.4	23.0	16.00	2330	34.14	53.8	70.0	74.8
Cleaner 2 Tail	13.2	10.3	4.92	557	5.93	7.4	7.5	5.8
Cleaner 1 Tail	85.0	66.6	3.98	259	3.05	38.7	22.5	19.3

Observations:

	Flotation in a 1.2 liter cell at 1650 RPM							
Cleaner 1 Conc	42.6	33.4	12.57	1780.6	25.40	61.3	77.5	80.7
Rougher Conc	127.6	100.0	6.85	767.0	10.51	100.0	100.0	100.0

RDi Project No.: **23009 Empire**
Flotation Test No. 19 **Date:** **21-Jul-23**

Purpose: Preliminary cleaner flotation test to recover a cleaned copper concentrate.

Sample: Approximately 130 grams of FT17 concentrate (Endo/Exo Master Composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
						PAX	AP208	AP404	MIBC	NaSH
Operations	min.	Solids %	pH-Start	pH-End	eH					
Pebble Mill Grinding	2.5	50		7.4	113	0	0	0	0	0
Conditioning	1.0	10	7.4	8.2	-100	77	154	154	77	1538
Cleaner 1 Flotation	6.0	10	8.2	8.2	105					
Conditioning	1.0	3	8.2	8.3	59	0	77	77	77	0
Cleaner 2 Flotation	2.0	3	8.3	8.1	109					
Total Reagent Used, g/mt of feed						77	231	231	154	1538
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	130							
Feed (calculated)	129.5	100.0	5.24	756	10.43	100.0	100.0	100.0
Cleaner 2 Conc	21.5	16.6	16.10	2560	39.05	51.0	56.2	62.2
Cleaner 2 Tail	9.2	7.1	6.63	563	7.22	9.0	5.3	4.9
Cleaner 1 Tail	98.8	76.3	2.75	382.0	4.50	40.0	38.5	32.9

Observations:

Flotation in a 1.2 liter cell at 1650 RPM

Cleaner 1 Conc	30.7	23.7	13.26	1961.6	29.51	60.0	61.5	67.1
Rougher Conc	129.5	100.0	5.24	756.5	10.43	100.0	100.0	100.0

RDi Project No.: **23009 Empire**
Flotation Test No. 20 **Date:** **11-Sep-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from various grades of sample.

Sample: Approximately 1 kg x 2 of 0.35 g/mt Au (#102 Endo/Exo variability composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	17.5	50		7.9/8.1	145/138	0	0	0	0	0
Conditioning	1.0	25	7.9/8.1	8.7/9.0	-54/-115	50	25	25	20	500
Ro Flotation - 1	3.0	25	8.7/9.0	8.6/8.9	17/-61					
Conditioning	1.0	25	8.6/8.9	8.9/9.1	-94/-131	25	25	25	10	63
Ro Flotation - 2	3.0	25	8.9/9.1	8.9/9.0	18/-73					
Conditioning	1.0	25	8.9/9.0	9.1/9.2	-106/-135	25	25	25	10	63
Ro Flotation - 3	3.0	25	9.1/9.2	9.0/9.1	-15/-79					
Conditioning	1.0	5	8.8	9.6	-162	8	15	15	10	150
Cleaner 1 Flotation	6.0	5	9.6	9.4	-66					
Total Reagent Used, g/mt of feed						100	75	75	40	625
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	2,000		0.35	19.9	0.50			
Feed (calculated)	1,894.7	100.0	0.28	17.6	0.47	100.0	100.0	100.0
Cleaner 1 Conc	15.5	0.8	17.9	802	2.04	52.2	37.3	3.6
Cleaner 1 Tail	48.0	2.5	0.71	53	1.43	6.4	7.6	7.8
Ro-Tail	1,831.2	96.6	0.12	10	0.43	41.4	55.0	88.6

Observations:

Ground in rod mill
Rougher flotation in a 3.2 liter cell at 1650 RPM
Cleaner flotation in a 1.2 liter cell at 1200 RPM

Rougher Conc	63.5	3.4	4.91	235.8	1.58	58.6	45.0	11.4

RDi Project No.: **23009 Empire**
Flotation Test No. 21 **Date:** **11-Sep-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from various grades of sample.

Sample: Approximately 1 kg x 2 of 0.5 g/mt Au (#101/103 Endo/Exo variability composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	17.5	50		8.4/8.4	103/97	0	0	0	0	0
Conditioning	1.0	25	8.4/8.4	9.5/9.5	-130/-137	50	25	25	20	375
Ro Flotation - 1	3.0	25	9.5/9.5	9.3/9.4	-11/-62					
Conditioning	1.0	25	9.3/9.4	9.7/9.7	-121/-161	25	25	25	10	63
Ro Flotation - 2	3.0	25	9.7/9.7	9.5/9.5	-36/-73					
Conditioning	1.0	25	9.5/9.5	9.8/9.8	-135/-170	25	25	25	10	63
Ro Flotation - 3	3.0	25	9.8/9.8	9.6/9.7	-49/-81					
Conditioning	1.0	2	9.2	9.7	-141	5	10	10	10	50
Cleaner 1 Flotation	6.0	2	9.7	9.3	-27					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	2,000		0.94	<7.0	0.37			
Feed (calculated)	1,917.4	100.0	0.97	8.8	0.34	100.0	100.0	100.0
Cleaner 1 Conc	6.3	0.3	137	381	6.22	46.4	14.2	6.0
Cleaner 1 Tail	20.3	1.1	3.39	51	1.37	3.7	6.1	4.3
Ro-Tail	1,890.8	98.6	0.49	7.1	0.31	49.9	79.6	89.8

Observations:

Ground in rod mill
Rougher flotation in a 3.2 liter cell at 1650 RPM
Cleaner flotation in a 1.2 liter cell at 1200 RPM

Rougher Conc	26.6	1.4	35.03	129.2	2.52	50.1	20.4	10.2

RDi Project No.: **23009 Empire**
Flotation Test No. 22 **Date:** **11-Sep-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from various grades of sample.

Sample: Approximately 1 kg x 2 of 0.8 g/mt Au (#298/667 Endo/Exo variability composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	17.5	50		8.4/8.4	78/91	0	0	0	0	0
Conditioning	1.0	25	8.4/8.4	9.3/9.4	-137/-151	50	25	25	20	375
Ro Flotation - 1	3.0	25	9.3/9.4	9.2/9.2	-46/-73					
Conditioning	1.0	25	9.2/9.2	9.4/9.4	-120/-147	25	25	25	10	63
Ro Flotation - 2	3.0	25	9.4/9.4	9.3/9.3	-46/-83					
Conditioning	1.0	25	9.3/9.3	9.5/9.5	-123/-150	25	25	25	10	63
Ro Flotation - 3	3.0	25	9.5/9.5	9.3/9.4	-55/-87					
Conditioning	1.0	7	9.1	10.2	-136	10	20	20	10	200
Cleaner 1 Flotation	6.0	7	10.2	10.0	-77					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	2,000		1.16	36.8	0.44			
Feed (calculated)	1,858.7	100.0	0.97	18.5	0.47	100.0	100.0	100.0
Cleaner 1 Conc	33.9	1.8	31.8	307	17.43	60.0	30.2	68.3
Cleaner 1 Tail	57.6	3.1	0.80	19	0.34	2.6	3.2	2.3
Ro-Tail	1,767.2	95.1	0.38	13	0.14	37.4	66.6	29.4

Observations:

Ground in rod mill
Rougher flotation in a 3.2 liter cell at 1650 RPM
Cleaner flotation in a 1.2 liter cell at 1200 RPM

Rougher Conc	91.5	4.9	12.29	125.7	6.67	62.6	33.4	70.6

RDi Project No.: **23009 Empire**
Flotation Test No. 23 **Date:** **11-Sep-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from various grades of sample.

Sample: Approximately 1 kg x 2 of 0.3 % Cu (#104/596 Endo/Exo variability composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	17.5	50		8.0/8.0	116/102	0	0	0	0	0
Conditioning	1.0	25	8.0/8.0	9.0/9.0	-155/-159	50	25	25	20	375
Ro Flotation - 1	3.0	25	9.0/9.0	8.9/8.9	-76/-113					
Conditioning	1.0	25	8.9/8.9	8.9/8.9	-155/-170	25	25	25	10	63
Ro Flotation - 2	3.0	25	8.9/8.9	9.0/8.9	-40/-90					
Conditioning	1.0	25	9.0/8.9	9.2/9.0	-156/-148	25	25	25	10	63
Ro Flotation - 3	3.0	25	9.2/9.0	9.1/9.0	-72/-81					
Conditioning	1.0	3	8.8	9.1	-124	5	20	20	10	50
Cleaner 1 Flotation	6.0	3	9.1	8.8	-30					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	2,000		0.09	<7.0	0.31			
Feed (calculated)	1,907.9	100.0	0.093	8.6	0.30	100.0	100.0	100.0
Cleaner 1 Conc	9.3	0.5	4.01	25	1.07	21.1	1.4	1.7
Cleaner 1 Tail	24.1	1.3	0.34	14	0.73	4.6	2.0	3.0
Ro-Tail	1,874.5	98.2	0.07	8.5	0.29	74.3	96.5	95.2

Observations:

Ground in rod mill
Rougher flotation in a 3.2 liter cell at 1650 RPM
Cleaner flotation in a 1.2 liter cell at 1200 RPM

Rougher Conc	33.4	1.8	1.36	17.1	0.83	25.7	3.5	4.8

RDi Project No.: **23009 Empire**
Flotation Test No. 24 **Date:** **11-Sep-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from various grades of sample.

Sample: Approximately 1 kg x 2 of 0.5% Cu (#291 Endo/Exo variability composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	17.5	50		7.9/7.8	121/125	0	0	0	0	0
Conditioning	1.0	25	7.9/7.8	8.4/8.4	-167/-132	50	25	25	20	500
Ro Flotation - 1	3.0	25	8.4/8.4	8.5/8.5	-84/-132					
Conditioning	1.0	25	8.5/8.5	8.6/8.6	-140/-181	25	25	25	10	63
Ro Flotation - 2	3.0	25	8.6/8.6	8.6/8.6	-74/-69					
Conditioning	1.0	25	8.6/8.6	8.7/8.6	-141/-123	25	25	25	10	63
Ro Flotation - 3	3.0	25	8.7/8.6	8.7/8.7	-81/-68					
Conditioning	1.0	7	8.6	8.8	-171	8	15	15	10	100
Cleaner 1 Flotation	6.0	7	8.8	8.7	-40					
Total Reagent Used, g/mt of feed						100	75	75	40	625
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	2,000		0.11	25.0	0.41			
Feed (calculated)	1,902.3	100.0	0.11	40	0.36	100.0	100.0	100.0
Cleaner 1 Conc	29.0	1.5	0.76	67	0.58	10.4	2.6	2.4
Cleaner 1 Tail	58.8	3.1	0.16	23	0.57	4.4	1.8	4.9
Ro-Tail	1,814.5	95.4	0.10	40	0.35	85.2	95.7	92.7

Observations:

Ground in rod mill
Rougher flotation in a 3.2 liter cell at 1650 RPM
Cleaner flotation in a 1.2 liter cell at 1200 RPM

Rougher Conc	87.8	4.6	0.36	37.5	0.57	14.8	4.3	7.3

RDi Project No.: **23009 Empire**
Flotation Test No. 25 **Date:** **11-Sep-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from various grades of sample.

Sample: Approximately 1 kg x 2 of 0.7 g/mt Au (#679/14 Endo/Exo variability composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	17.5	50		8.2/8.4	95/67	0	0	0	0	0
Conditioning	1.0	25	8.2/8.4	9.2/9.3	-141/-174	50	25	25	20	375
Ro Flotation - 1	3.0	25	9.2/9.3	9.0/9.1	-74/-91					
Conditioning	1.0	25	9.0/9.1	9.2/9.3	-144/-151	25	25	25	10	63
Ro Flotation - 2	3.0	25	9.2/9.3	9.2/9.2	-48/-92					
Conditioning	1.0	25	9.2/9.2	9.4/9.3	-173/-147	25	25	25	10	63
Ro Flotation - 3	3.0	25	9.4/9.3	9.4/9.2	-82/-91					
Conditioning	1.0	8	9.0	9.8	-167	10	20	20	10	100
Cleaner 1 Flotation	6.0	8	9.8	9.6	-56					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	2,000		0.34	10.3	0.58			
Feed (calculated)	1,904.6	100.0	0.31	13.9	0.48	100.0	100.0	100.0
Cleaner 1 Conc	17.9	0.9	14.2	398	2.81	43.0	26.9	5.6
Cleaner 1 Tail	79.9	4.2	0.38	16	0.84	5.1	4.8	7.4
Ro-Tail	1,806.8	94.9	0.17	10	0.44	51.9	68.3	87.1

Observations:

Ground in rod mill
Rougher flotation in a 3.2 liter cell at 1650 RPM
Cleaner flotation in a 1.2 liter cell at 1200 RPM

Rougher Conc	97.8	5.1	2.91	85.9	1.20	48.1	31.7	12.9

RDi Project No.: **23009 Empire**
Flotation Test No. 26 **Date:** **11-Sep-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from various grades of sample.

Sample: Approximately 1 kg x 2 of 0.15 g/mt Au (#127/128 Endo/Exo variability composite).
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	17.5	50		7.8/7.9	102/94	0	0	0	0	0
Conditioning	1.0	26	7.8/7.9	9.0/9.0	-116/-111	50	25	25	20	375
Ro Flotation - 1	3.0	26	9.0/9.0	8.8/8.7	-120/-147					
Conditioning	1.0	26	8.8/8.7	9.0/9.0	-106/-145	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.0/9.0	8.9/8.9	-34/-43					
Conditioning	1.0	26	8.9/8.9	9.2/9.0	-161/-137	25	25	25	10	63
Ro Flotation - 3	3.0	26	9.2/9.0	9.1/8.9	-52/-47					
Conditioning	1.0	2	8.8	9.3	-190	10	20	20	10	150
Cleaner 1 Flotation	6.0	2	9.3	9.0	-55					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	2,000		0.16	<7.0	0.29			
Feed (calculated)	1,988.1	100.0	0.18	5.6	0.25	100.0	100.0	100.0
Cleaner 1 Conc	8.1	0.4	18.4	145	6.95	41.6	10.5	11.3
Cleaner 1 Tail	18.8	0.9	0.71	13	0.63	3.7	2.2	2.4
Ro-Tail	1,961.2	98.6	0.10	5.0	0.22	54.7	87.4	86.3

Observations:

Ground in rod mill
Rougher flotation in a 3.2 liter cell at 1650 RPM
Cleaner flotation in a 1.2 liter cell at 1200 RPM

Rougher Conc	26.9	1.4	6.04	52.7	2.53	45.3	12.6	13.7

RDi Project No.: **23009 Empire**
Flotation Test No. 27 **Date:** **11-Sep-23**

Purpose: Preliminary flotation test to recover a bulk concentrate from various grades of sample.

Sample: Approximately 1 kg x 2 of high grade mag composite.
Approximately P_{80} of 100 mesh

Procedure:

	Conditions					Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	eH	PAX	AP208	AP404	MIBC	NaSH
Rod Mill Grinding	24.8	50		7.1/7.6	100/16	0	0	0	0	0
Conditioning	1.0	26	7.1/7.6	8.9/8.1	-175/-152	50	25	25	20	375
Ro Flotation - 1	3.0	26	8.9/8.1	8.9/8.4	-119/-92					
Conditioning	1.0	26	8.9/8.4	9.0/8.5	-183/-152	25	25	25	10	63
Ro Flotation - 2	3.0	26	9.0/8.5	8.9/8.6	-106/-97					
Conditioning	1.0	26	8.9/8.6	9.0/8.7	-159/-141	25	25	25	10	63
Ro Flotation - 3	3.0	26	9.0/8.7	8.9/8.7	-104/-93					
Conditioning	1.0	1	8.7	9.2	-221	10	20	20	10	150
Cleaner 1 Flotation	6.0	1	9.2	8.8	-91					
Total Reagent Used, g/mt of feed						100	75	75	40	500
Solution Concentration (% or g/drop)						1.0%	1.0%	1.0%	0.0050	5.0%

Results:

Products	Weight		Chemical Analysis			Percent Distribution		
	gr.	%	Au g/mt	Ag g/mt	Cu %	Au	Ag	Cu
Feed (analyzed)	2,000		2.47	10.5	0.13			
Feed (calculated)	1,999.8	100.0	2.81	8.6	0.12	100.0	100.0	100.0
Cleaner 1 Conc	3.6	0.2	1129	350	1.61	72.4	7.3	2.4
Cleaner 1 Tail	9.2	0.5	8.43	15	0.39	1.4	0.8	1.5
Ro-Tail	1,987.0	99.4	0.74	8.0	0.12	26.2	91.9	96.1

Observations:

Ground in rod mill
Rougher flotation in a 3.2 liter cell at 1650 RPM
Cleaner flotation in a 1.2 liter cell at 1200 RPM

Rougher Conc	12.8	0.6	324	109.2	0.73	73.8	8.1	3.9

APPENDIX H: LOCKED-CYCLE TEST RESULTS

EM LCT-1 **Project No:** 23009 Empire **Date**: 17-Sep

Purpose: Complete locked-cycle test to determine ultimate recovery and concentrate grade.

Procedure: As outlined below.

Feed: **Endo/Exo Master Composite, 10-kg x 6 of minus 6 mesh for rougher flotation.**

Grind: 22 minutes/10-kg at 50% solids in Abbey rod mill Primary P_{80} = 150 µm

Conditions:

Stage	Reagents, g/mt based on rougher flotation feed					Time, minutes				
	PAX	AP404	AP208	NaSH	MIBC	Grind	Cond.	Froth	Start pH	End pH
Grind	-	-	-		-	22.0				8.2
Rougher Flotation - 1	50	25	25	375	4		1	5	9.0	9.0
Rougher Flotation - 2	25	25	25	63	2		1	5	9.1	9.0
Rougher Flotation - 3	25	25	25	63	2		1	5	9.2	9.0
Rougher Flotation - 4	25	25	25	63	2		1	5	9.1	9.0
1st Cleaner	0.17	0.35	0.35	1.25	0.17		3	6	9.2	9.0
Total:	125	100	100	565	10	22	7	26		

Stage	Rougher	1st Cleaner
Flotation Cell	Cubic Foot cell	250-g
Speed rpm	1650	1200

Product	Weight		Assays Values			% Distribution		
	Dry	%	Au (g/mt)	Ag (g/mt)	Cu (%)	Au	Ag	Cu
1st Cl Conc	**39.5**	**0.4**	**17.43**	**2288**	**35.7**	**39.3**	**36.2**	**33.5**
1st Cleaner Tail	36.7	0.1	2.184	203	1.8	0.8	0.5	0.3
Ro Tail	9885.0	99.6	0.11	15.94	0.282	60.0	63.3	66.2
Head (calculated)	9922	100	0.18	25.3	0.43	100	100	100

Product	Weight		Assay Values			% Distribution		
	Dry	%	Au (g/mt)	Ag (g/mt)	Cu (%)	Au	Ag	Cu
1st Cl Conc A	46.0	0.08	11.8	1878	30.4	5.1	5.5	5.6
1st Cl Conc B	42.5	0.07	13.9	2038	27.2	5.5	5.5	4.6
1st Cl Conc C	41.2	0.07	13.7	2011	28.3	5.3	5.2	4.6
1st Cl Conc D	41.0	0.07	16.8	2389	38.8	6.5	6.2	6.3
1st Cl Conc E	35.5	0.06	17.1	2350	32.8	5.7	5.3	4.6
1st Cl Conc F	41.9	0.07	18.5	2158	35.6	7.3	5.7	5.9
1st Cl Tail F	36.7	0.06	2.19	204	1.8	0.8	0.5	0.3
Rougher Tail A	9866	16.56	0.10	24	0.28	9.2	15.0	11.1
Rougher Tail B	9875	16.58	0.08	14	0.30	7.4	8.7	11.7
Rougher Tail C	9885	16.59	0.19	20	0.30	17.6	12.5	11.7
Rougher Tail D	9870	16.57	0.12	19	0.27	11.1	11.9	10.6
Rougher Tail E	9893	16.61	0.09	14	0.29	8.3	8.8	11.4
Rougher Tail F	9891	16.61	0.11	15	0.29	10.2	9.4	11.4
Head (calc.)	59565	100	0.18	27	0.42	100	100	100
(direct)			0.15	24	0.40			

Stability

	Weight		Assay Values		
	g	%	Au (g/mt)	Ag (g/mt)	Cu (%)
Total In All Cycles	59565	100	0.18	27	0.42
Avg In Per Cycle	9927	16.7			

Total Products Out Per Cycle	Weight	Units out as a % of Units in/Cycle

Normolize Wts

	g	Wt %	Au	Ag	Cu
Cycle A	9912	99.8	86.0	122.5	100.2
Cycle B	9917	99.9	77.7	85.3	97.6
Cycle C	9926	100.0	137.4	106.4	98.1
Cycle D	9911	99.8	105.4	108.3	101.6
Cycle E	9929	100.0	84.2	84.2	96.5
Cycle F	9933	100.1	104.8	90.6	104.4

Average of A-C	99.9	100.4	104.7	98.6
Average of C-D	99.9	121.4	107.3	99.9
Average of D-E	99.9	94.8	96.2	99.1
Average of D-F	100.0	98.1	94.3	100.8
Average of E-F	100.0	94.5	87.4	100.5
Average of C-F	100.0	107.9	97.3	100.2

100.0

Cycle Statistics (Least Squares)

0.5	0.5	1.0
457.1	460.9	917.9
27.2	26.4	53.6
3.6	3.5	7.2
30.1	30.5	60.6
63.2	63.6	126.8

LCT Wet Product Weights

	Cycle 1	Cycle 2	Cycle 3	Cycle 4	Cycle 5	Cycle 6
Rougher Conc	161.9	146.3	140.7	150.1	119.6	116
CL 1 Con	58.4	55.2	54	53	47.5	53.6
CL 1 Tail	97.9	88.4	74.1	98.2	69.4	61.6

EM LCT-1 **Project No:** 23009 Empire **Date**: 17-Sep

Purpose: Complete locked-cycle test to determine ultimate recovery and concentrate grade.

Procedure: As outlined below.

Feed: **Mag Master Composite, 10-kg x 6 of minus 6 mesh for rougher flotation.**

Grind: 22 minutes/10-kg at 50% solids in Abbey rod mill Primary P_{80} = 150 µm

Conditions:

Stage	Reagents, g/mt based on rougher flotation feed					Time, minutes			Pu	
	PAX	AP404	AP208	NaSH	MIBC	Grind	Cond.	Froth	Start pH	End pH
Grind	-	-	-		-	19.5				7.8
Rougher Flotation - 1	50	25	25	500	4		1	5	9.0	8.9
Rougher Flotation - 2	25	25	25	63	2		1	5	9.0	8.8
Rougher Flotation - 3	25	25	25	63	2		1	5	9.0	8.8
Rougher Flotation - 4	25	25	25	63	2		1	5	9.0	8.9
1st Cleaner	0.17	0.35	0.35	1.25	0.17		3	6	9.3	8.9
Total:	125	100	100	690	10	19.5	7	26		

Stage	Rougher	1st Cleaner
Flotation Cell	Cubic Foot cell	250-g
Speed rpm	1650	1200

Product	Weight		Assays Values			% Distribution		
	Dry	%	Au (g/mt)	Ag (g/mt)	Cu (%)	Au	Ag	Cu
1st Cl Conc	**37.5**	**0.4**	**10.68**	**563**	**6.0**	**20.9**	**11.6**	**3.8**
1st Cleaner Tail	51.3	0.1	0.628	47	1.0	0.3	0.2	0.2
Ro Tail	9872.5	99.5	0.15	16.29	0.572	78.8	88.2	96.0
Head (calculated)	9924	100	0.19	18.5	0.60	100	100	100

Product	Weight		Assay Values			% Distribution		
	Dry	%	Au (g/mt)	Ag (g/mt)	Cu (%)	Au	Ag	Cu
1st Cl Conc A	29.7	0.05	14.2	599	7.97	3.6	1.8	0.6
1st Cl Conc B	31.0	0.05	15.7	633	7.93	4.1	2.0	0.7
1st Cl Conc C	26.5	0.04	15.4	760	8.23	3.5	2.1	0.6
1st Cl Conc D	35.0	0.06	14.1	748	6.63	4.2	2.7	0.6
1st Cl Conc E	38.0	0.06	6.76	418	4.11	2.2	1.6	0.4
1st Cl Conc F	39.6	0.07	11.5	543	7.19	3.9	2.2	0.8
1st Cl Tail F	51.3	0.09	0.63	47	1.05	0.3	0.2	0.1
Rougher Tail A	9881	16.60	0.17	11	0.69	14.3	11.2	18.3
Rougher Tail B	9892	16.62	0.16	14	0.57	13.5	14.2	15.3
Rougher Tail C	9887	16.61	0.14	12	0.64	11.8	12.2	16.9
Rougher Tail D	9890	16.61	0.16	17	0.56	13.5	17.3	15.0
Rougher Tail E	9849	16.55	0.10	14	0.56	8.4	14.2	14.8
Rougher Tail F	9878	16.59	0.20	18	0.60	16.8	18.3	15.9
Head (calc.)	59528	100	0.20	16	0.62	100	100	100
(direct)			0.23	18	0.67			

Stability

	Weight		Assay Values		
	g	%	Au (g/mt)	Ag (g/mt)	Cu (%)
Total In All Cycles	59528	100	0.20	16	0.62
Avg In Per Cycle	9921	16.7			

Total Products Out Per Cycle	Weight	Units out as a % of Units in/Cycle

Normolize Wts

	g	Wt %	Au	Ag	Cu
Cycle A	9911	99.9	107.4	78.0	113.4
Cycle B	9923	100.0	105.7	97.5	95.5
Cycle C	9913	99.9	91.6	85.6	105.0
Cycle D	9925	100.0	106.1	119.8	93.6
Cycle E	9887	99.7	63.4	94.8	91.5
Cycle F	9918	100.0	124.2	122.9	100.1

Average of A-C	99.9	101.6	87.0	104.6
Average of C-D	100.0	98.8	102.7	99.3
Average of D-E	99.8	84.7	107.3	92.5
Average of D-F	99.9	97.9	112.5	95.1
Average of E-F	99.8	93.8	108.8	95.8
Average of C-F	99.9	96.3	105.8	97.5

99.9

Cycle Statistics (Least Squares)

6.7	7.0	13.8
1.5	1.5	3.0
243.7	238.6	482.3
9.3	8.6	17.9
41.7	39.1	80.8
14.8	13.9	28.7

LCT Wet Product Weights

	Cycle 1	Cycle 2	Cycle 3	Cycle 4	Cycle 5	Cycle 6
Rougher Conc	144.4	126.9	136.3	127.5	198.6	145.1
CL 1 Con	46.8	47.5	40.7	56.4	61.4	61.8
CL 1 Tail	93	74.8	89.6	70.9	131.3	82.1

APPENDIX I: COPPER CONCENTRATE ANALYSES

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc
11475 West I-70 Frontage Road North
Wheat Ridge, CO 80033
Attention: Deepak Malhotra & Ryan Allen

Laboratory No.: 231931
Client Number: F174
Date Received: 05 Oct 2023
Date Completed: 16 Oct 2023

Method: 4 Acid digestion, ICP analysis, Ag by AA analysis.

Lab code: 7060

Element:	Ag	Al	Ba	Ca	Cd	Co	Cr	Cu	Fe	K	Mg	Mn	Mo	Na	Ni	Pb	Sr	Ti	V	W	Zn
Detection Limit:	0.2	0.01	1	0.01	1	1	1	0.01	0.01	0.01	0.01	0.01	0.01	0.01	5	0.01	5	0.01	1	10	0.01
Reporting Unit:	mg/kg	%	mg/kg	%	mg/kg	mg/kg	mg/kg	%	%	%	%	%	%	%	mg/kg	%	mg/kg	%	mg/kg	mg/kg	%
23009 LCT1 CL Con Cyc 6	1936.0	0.93	244	2.28	43	132	43	34.61	13.81	0.32	1.30	0.17	0.16	0.16	165	0.26	48	0.04	37	604	0.40
23009 LCT2 CL Con Cyc 6	514.0	4.70	751	3.01	33	140	92	6.52	16.61	0.56	1.21	1.31	0.07	0.07	109	0.11	117	0.20	68	497	0.83

Karen Boldi, Laboratory Technician

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc
11475 West I-70 Frontage Road North
Wheat Ridge, CO 80033
Attention: Deepak Malhotra & Ryan Allen

Laboratory No.: 231931
Client Number: F174
Date Received: 05 Oct 2023
Date Completed: 15 Nov 2023

Method: 2-Acid digestion ICP analysis for As, Sb, Bi, Se, Te & CVAA for Hg.
Lab code: 7060

Element:	Arsenic	Bismuth	Antimony	Mercury	Selenium	Tellurium
Detection Limit:	2	2	2	0.02	5	2
Units:	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg
23009 LCT1 CL Con Cyc 6	2937	3544	47	16.70	298	388
23009 LCT2 CL Con Cyc 6	841	398	138	66.10	33	88

Karen Boldi

Karen Boldi, Laboratory Technician

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc
11475 West I-70 Frontage Road North
Wheat Ridge, CO 80033
Attention: Deepak Malhotra & Ryan Allen

Laboratory No.: 231931
Client Number: F174
Date Received: 05 Oct 2023
Date Completed: 16 Nov 2023

Method: Lithium metaborate fusion, ICP analysis.

Lab Code: 7046

Element:	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	Na_2O	K_2O	TiO_2	MnO	SrO	BaO	Cr_2O_3	P_2O_5	$LOI_{1000\,^oC}$	SUM
Detection Limit:	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.10	
Sample ID:	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%
23009 LCT1 CL Con Cyc 6	20.07	1.87	18.86	3.20	2.25	0.10	0.30	0.07	0.24	0.00	0.03	0.01	0.41	13.81	61.21
23009 LCT2 CL Con Cyc 6	41.29	8.91	22.97	4.10	2.00	0.08	0.64	0.35	1.70	0.01	0.08	0.01	0.83	13.26	96.24

Karen Boldi

Karen Boldi, Laboratory Technician

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc
11475 West I-70 Frontage Road North
Wheat Ridge, CO 80033
Attention: Deepak Malhotra & Ryan Allen

Laboratory No.: 231931
Client Number: F174
Date Received: 05 Oct 2023
Date Completed: 15 Nov 2023

Method: Nitric acid digestion for Chloride & sodium peroxide fusion for Fluoride, ion select electrode analysis.

Lab code:	7055	7051
Element:	Chloride	Fluoride
Detection Limit:	20	10
Units:	mg/kg	mg/kg
23009 LCT1 CL Con Cyc 6	19600	204
23009 LCT2 CL Con Cyc 6	104	500
Blank	26	<10
OREAS 701; STSD-1	216	1000
Certified Reference Material	**Cl^-**	**F^-**
OREAS 701	227	
95% Confidence Interval of Mean	69	
STSD-1		950

Karen Boldi

Karen Boldi, Laboratory Technician

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

APPENDIX J: THICKENING TEST RESULTS

Settling Test No. 1

Project: **Empire**
Date: **9/28/2023**

PURPOSE: To determine the theoretical settling area needed to thicken the sample at a feed pulp density of approximately 25% solids with 17 ppm flocculant. The flocculent used was a high weight molecular anionic polymer supplied by Polymer Ventures (DAF-10).

SAMPLE: Endo/Exo Master Composite Flotation Tails at P_{80} 100 mesh
The pH was not adjusted and the test was completed at room temperature.

		WEIGHT	FINAL SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL
		% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS
A	GRADUATE CYLINDER LEVEL, ML	1000	335	550	468	402	348
B	GROSS WEIGHT GR. (cylinder + slurry)	1486					
C	TARE WEIGHT GR. (weight of empty cylinder)	284					
D	NET WEIGHT GR. **(B - C)**	1202					
E	SP. GR. PULP **(A ÷D)**	1.20					
F	SP. GR. OF SOLIDS (measured)	2.84					
G	PERCENT SOLIDS V/V **(I ÷F) ÷A**	10%					
H	PERCENT SOLIDS W/W **(I ÷D)**	25%	**56%**	**40%**	**45%**	**50%**	**55%**
I	WEIGHT OF SOLIDS GR. (measured)	297					
J	Starting pH	7.9					
K	Flocculent Strength (%)	0.1%					
L	Flocculent, cc	5.0					
M	Final Turbidity, NTU	57.6					
	Critical Time, Hours		24.000	0.008	0.016	0.039	0.338
	Unit Area, m^2/mt/day		2.405	0.001	0.002	0.004	0.034
	Unit Area, ft2/st/day		23.469	0.008	0.015	0.038	0.330

Cylinder Height, Distance for 1 liter = 346 mm.

Settling Test No. 2

Project: **Empire**
Date: **9/28/2023**

PURPOSE: To determine the theoretical settling area needed to thicken the sample at a feed pulp density of approximately 25% solids with 17 ppm flocculant. The flocculent used was a high weight molecular anionic polymer supplied by Polymer Ventures (DAF-10).

SAMPLE: Mag Master Composite Flotation Tails at P_{80} 100 mesh
The pH was not adjusted and the test was completed at room temperature.

		WEIGHT	FINAL SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL
		% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS
A	GRADUATE CYLINDER LEVEL, ML	1000	351	557	473	406	352
B	GROSS WEIGHT GR. (cylinder + slurry)	1494					
C	TARE WEIGHT GR. (weight of empty cylinder)	285					
D	NET WEIGHT GR. **(B - C)**	1209					
E	SP. GR. PULP **(A ÷D)**	1.21					
F	SP. GR. OF SOLIDS (measured)	2.85					
G	PERCENT SOLIDS V/V **(I ÷F) ÷A**	11%					
H	PERCENT SOLIDS W/W **(I ÷D)**	25%	**55%**	**40%**	**45%**	**50%**	**55%**
I	WEIGHT OF SOLIDS GR. (measured)	301					
J	Starting pH	7.9					
K	Flocculent Strength (%)	0.1%					
L	Flocculent, cc	5.0					
M	Final Turbidity, NTU	202					
	Critical Time, Hours		24.000	0.016	0.036	0.092	6.550
	Unit Area, m^2/mt/day		2.397	0.002	0.004	0.009	0.654
			23.393	0.016	0.035	0.089	6.384

Cylinder Height, Distance for 1 liter = 345 mm.

Settling Test No. 3

Project: **Empire**
Date: **10/12/2023**

PURPOSE: To determine the theoretical settling area needed to thicken the sample at a feed pulp density of approximately 40% solids with coagulant and flocculant. The coagulant and flocculent used were non-ionic polymer (PV-1207 and MagnaFloc-333).

SAMPLE: Endo/Exo Master Composite Acid Leach Residue at P_{80} 100 mesh
The pH was not adjusted and the test was completed at room temperature.

		WEIGHT	FINAL SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL
		% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS
A	GRADUATE CYLINDER LEVEL, ML	1000	573			746	640
B	GROSS WEIGHT GR. (cylinder + slurry)	1639					
C	TARE WEIGHT GR. (weight of empty cylinder)	284					
D	NET WEIGHT GR. **(B - C)**	1355					
E	SP. GR. PULP **(A ÷D)**	1.35					
F	SP. GR. OF SOLIDS (measured)	2.89					
G	PERCENT SOLIDS V/V **(I ÷F) ÷A**	16%					
H	PERCENT SOLIDS W/W **(I ÷D)**	35%	**54%**			**45%**	**50%**
I	WEIGHT OF SOLIDS GR. (measured)	476					
J	Starting pH	7.9					
K	Coagulant Strength (%)	1.0%					
L	Coagulant, cc	1.0					
M	Flocculent Strength (%)	0.1%					
N	Flocculent, cc	5.0					
	Critical Time, Hours		24.000			0.916	3.243
	Unit Area, m^2/mt/day		2.146			0.082	0.290
	Unit Area, ft2/st/day		20.940			0.799	2.829

Cylinder Height, Distance for 1 liter = 344 mm.

Settling Test No. 4

Project: **Empire**
Date: **10/12/2023**

PURPOSE: To determine the theoretical settling area needed to thicken the sample at a feed pulp density of approximately 40% solids with coagulant and flocculant. The coagulant and flocculent used were non-ionic polymer (PV-1207 and MagnaFloc-333).

SAMPLE: Mag Master Composite Acid Leach Residue at P_{80} 100 mesh
The pH was not adjusted and the test was completed at room temperature.

		WEIGHT	FINAL SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL	CALCULATED SETTLED VOL
		% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS	% SOLIDS
A	GRADUATE CYLINDER LEVEL, ML	1000	515			722	619
B	GROSS WEIGHT GR. (cylinder + slurry)	1643					
C	TARE WEIGHT GR. (weight of empty cylinder)	285					
D	NET WEIGHT GR. **(B - C)**	1358					
E	SP. GR. PULP **(A ÷D)**	1.36					
F	SP. GR. OF SOLIDS (measured)	3.06					
G	PERCENT SOLIDS V/V **(I ÷F) ÷A**	15%					
H	PERCENT SOLIDS W/W **(I ÷D)**	34%	**56%**			**45%**	**50%**
I	WEIGHT OF SOLIDS GR. (measured)	466					
J	Starting pH	7.9					
K	Coagulant Strength (%)	1.0%					
L	Coagulant, cc	1.0					
M	Flocculent Strength (%)	0.1%					
N	Flocculent, cc	5.0					
	Critical Time, Hours		24.000			0.232	0.573
	Unit Area, m^2/mt/day		2.140			0.021	0.051
			20.888			0.202	0.498

Cylinder Height, Distance for 1 liter = 344 mm.

APPENDIX K: FILTRATION TEST RESULTS

Filtration Test Results

RDI Project: **23009 Empire**
Date: **29-Sep-23**

Sample: Leach residues at P80 100 mesh

Purpose: To examine the vacuum and pressure filtering characteristics of the samples

GENERAL				SLURRY	VACUUM	PRESSURE	FORM TIME	DRY TIME	FILTRATE	CAKE					RESULTS		
TEST NO.	Date	Sample	Particle Size (P_{80})	% SOLIDS	IN. HG.	PSI	Minutes	Minutes	ML	THICKNESS, MM	CRACKS	WET WEIGHT CAKE-GRAMS	DRY WEIGHT CAKE-GRAMS	% CAKE MOISTURE	CYCLE, HR	DRY /LB.SQ.FT./HR	GALLONS/ SQ.FT./HR
1	9/29/2023	Endo/Exo Master Comp Leach Residue	100 mesh	38%	16.5	-----	1.1	0	123	4	N	127.4	96.2	24.5%	0.019	128.7	19.72
2	9/29/2023	Endo/Exo Master Comp Leach Residue	100 mesh	41%	-----	80	0.7	0	508	6	N	478.0	400.6	16.2%	0.011	911.3	138.49
3	9/29/2023	Endo/Exo Master Comp Leach Residue	100 mesh	40%	-----	80	0.8	0	777	9	N	705.2	594.4	15.7%	0.014	1103.8	172.92
4	9/29/2023	Mag Master Comp Leach Residue	200 mesh	39%	16.5	-----	1.6	0	118	4	N	131.3	97.3	25.9%	0.026	95.2	13.84
5	9/29/2023	Mag Master Comp Leach Residue	200 mesh	39%	-----	80	1.1	0	547	6	N	476.9	397.9	16.6%	0.018	574.7	94.68
6	9/29/2023	Mag Master Comp Leach Residue	200 mesh	39%	-----	80	3.3	0	794	9	N	715.4	592.5	17.2%	0.055	272.3	43.73

APPENDIX L: COPPER CEMENTATION TEST RESULTS

Project: 23009 Empire
Date: 26-Oct-23

Copper Cementation Test 1

Purpose: To determine the cementation of copper from solution by addition of iron chips

Sample: 1000 mL of Endo/Exo sulfuric acid leachate (from acid leaching test 35)

Procedure: at ambient temperature with stirring, approximately 4 grams of high purity iron filings (40-60 mesh) were added to the solution
after one half hour, an additional 2 grams of iron filings were added.
after 2.5 hours, the solution had changed from green to gray/colorless.
the solution was filtered, the solids washed and the initial solution, final solution, and residues were submitted for ICP-OES analysis

Conditions:

Sample Solution	Leach time	Iron Additions
1029 g	2.5 hrs	6.009 g

SG Initial 1.029

Summary of Results

Parameter	Cu
Cu Reduction in solution (%)	94.4

A. Leach Conditions

Time	solution wt	solution vol	Reagents Added g	
hr	g	ml	Iron Filings	pH
0	1029	1000	4.008	1.50
0.5			2.001	
2.5				
total			6.009	

SG Final 1.0348

B. Products and Analysis

	Weight	Volume	Cu		Fe	
Leach Product	g	ml	wt %	mg/L	wt %	mg/L

Feed Solution (analyzed)	1029	1000		1607.6	3542.3
Final Solution (analyzed)				90.6	6210.9
cement copper (analyzed)	6.301		28.7	41.2	

feed solution g Cu	1.607
Final Sol'n g Cu	0.091
cement copper g	1.516
Cu in cement copper (assayed) g	1.808
Cu accounted for by mass	116%

Sample Id	Project ID	Sample Name	Acquisition Time	Ag (ppm)	Al (ppm)	As (ppm)	Ba (ppm)	Be (ppm)	Ca (ppm)	Cd (ppm)	Co (ppm)	Cr (ppm)	Cu (ppm)	Fe (ppm)	K (ppm)	Mg (ppm)	Mn (ppm)	Mo (ppm)	Na (ppm)	Ni (ppm)	P (ppm)	Pb (ppm)	S (ppm)	Sb (ppm)	Se (ppm)	Sn (ppm)	Sr (ppm)	Th (ppm)	Ti (ppm)	Tl (ppm)	U (ppm)	V (ppm)	Zn (ppm)
AN23-3493	23009	Endo/Exo Cem 0 hr	11/21/2023 11:21:11 AM	BD	1158	7.44	BD	BD	498	3.58	8.22	BD	1608	3542	60.4	1021	891	BD	142	5.10	364	3.16	7248	BD	BD	BD	4.23	BD	BD	BD	8.26	BD	1351
AN23-3494	23009	Endo/Exo Cem 2.5 hr	11/21/2023 10:45:57 AM	BD	1244	BD	5.07	BD	631	3.11	8.98	4.10	80.6	6211	60.4	973	847	BD	92.4	6.49	75.9	BD	7458	BD	BD	BD	4.89	BD	BD	BD	16.0	BD	1275

AN23-3491	23009	EH Cu Cementation residue	11/20/2023 3:14:22 PM	BD	2604	947	456	BD	1307	157	156	918	286781	411528	602	342	2237	327	BD	412	23420	534	10122	BD	BD	224	BD	193	489	17.7	1580	84.8	388
			reported as % wt	BD	0.26	0.08	0.05	BD	0.13	0.02	0.02	0.09	28.7	41.2	0.06	0.03	0.22	0.03	BD	0.04	2.34	0.05	1.01	BD	BD	0.02	BD	0.02	0.05	0.00	0.16	0.01	0.04

APPENDIX M: DISTRIBUTION OF GOLD IN FLOTATION TAILING

Project: **23009 Empire**
Date: **20-Jul-23**

Screen Assay Test No. 1 a

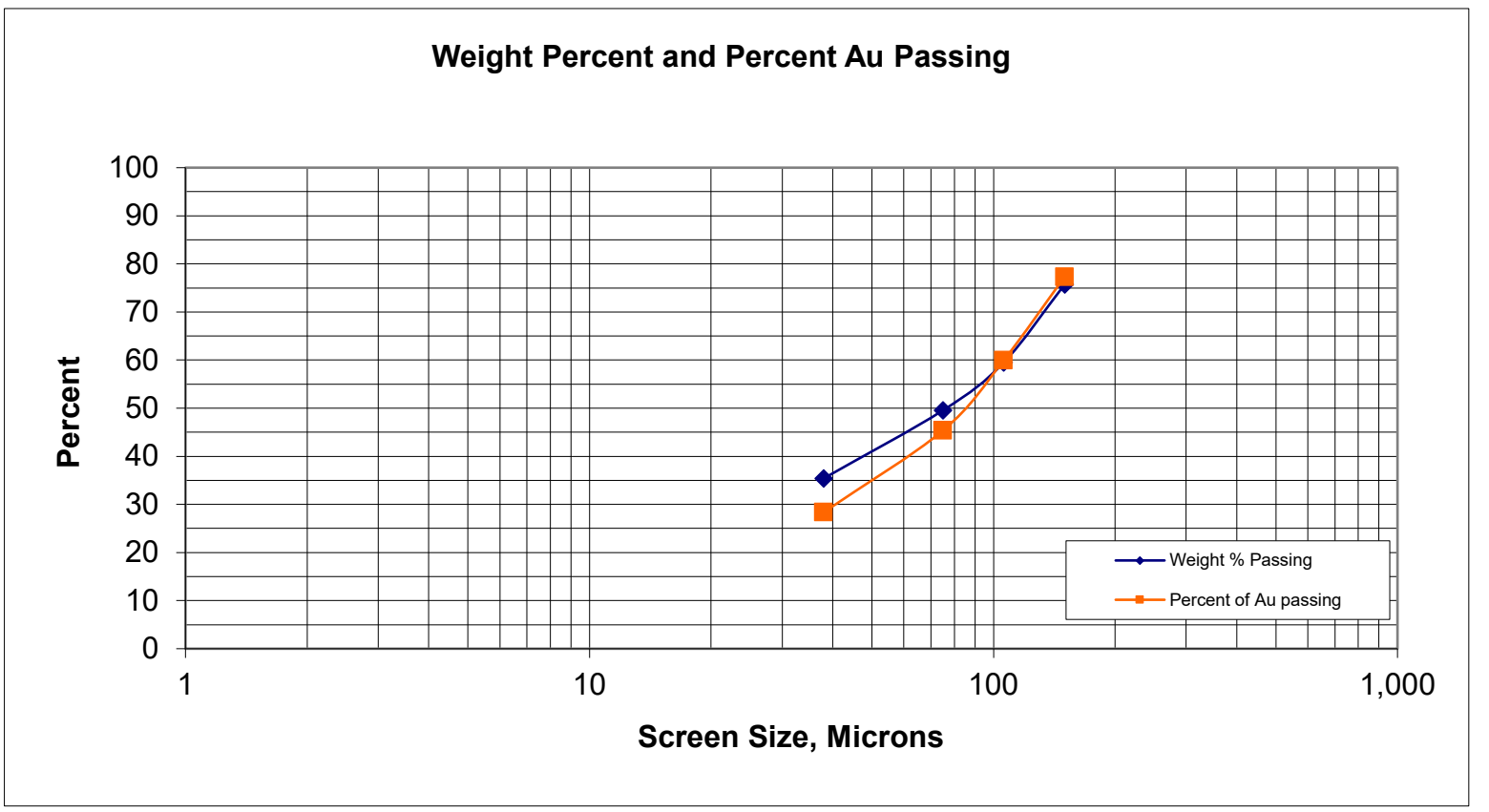
Purpose: To determine the distribution of gold in the sample.

Sample: Approximately 1230 grams of FT 17 Tails (Endo/Exo Master Composite).

Procedure: The sample was screened with screens ranging from 100 mesh to 400 mesh. A split of the minus 10 mesh material was screened
30-minutes (Ro-Tap). The screen fractions were weighed and a representative sample from each fraction was submitted for assa

Results:

Product		Weight						
Fraction (Retained)	microns	grams	Direct %	Cumm % Retained	Analysis Au(g/mt)	% Dist. Au	Cumm % Au Passing	Assay, Au (g/mt) in screen undersize
Feed (Calculated)		1,232.6	100.0		0.07	100.0		
+100 mesh	150	299.3	24.3	24.3	0.07	22.7	77.3	0.08
150 mesh x 100 mesh	106	200.0	16.2	40.5	0.08	17.3	60.0	0.08
200 mesh x 150 mesh	75	122.4	9.9	50.4	0.11	14.6	45.4	0.07
400 mesh x 200 mesh	38	174.5	14.2	64.6	0.09	17.0	28.4	0.06
- 400 mesh	-38	436.4	35.4	100.0	0.06	28.4		



Project: **23009 Empire**
Date: **20-Jul-23**

Screen Assay Test No. 1 b

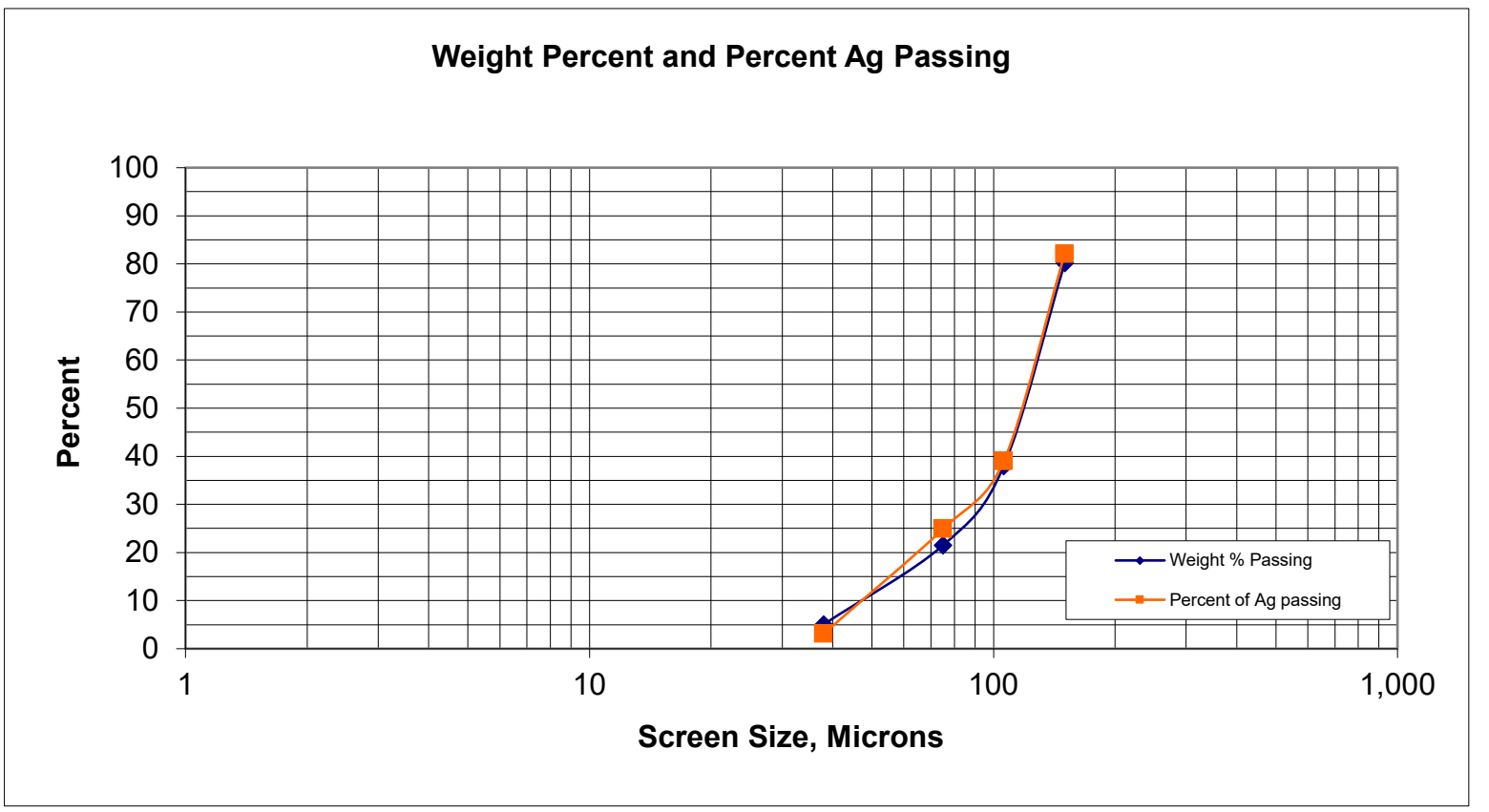
Purpose: To determine the distribution of silver in the sample.

Sample: Approximately 1230 grams of FT 17 Tails (Endo/Exo Master Composite).

Procedure: The sample was screened with screens ranging from 100 mesh to 400 mesh. A split of the minus 10 mesh material was screened
30-minutes (Ro-Tap). The screen fractions were weighed and a representative sample from each fraction was submitted for assa

Results:

Product		Weight						
Fraction (Retained)	microns	grams	Direct %	Cumm % Retained	Analysis Ag(g/mt)	% Dist. Ag	Cumm % AgPassing	Assay, Ag (g/mt) in screen undersize
Feed (Calculated)		443.3	100.0		11.3	100.0		
+100 mesh	150	88.0	19.9	19.9	10.2	18.0	82.0	11.54
150 mesh x 100 mesh	106	187.0	42.2	62.0	11.5	43.0	39.0	11.58
200 mesh x 150 mesh	75	73.0	16.5	78.5	9.7	14.1	24.9	13.05
400 mesh x 200 mesh	38	73.0	16.5	95.0	14.9	21.8	3.1	7.00
- 400 mesh	-38	22.4	5.0	100.0	7.0	3.1		



Project: **23009 Empire**
Date: **20-Jul-23**

Screen Assay Test No. 1 c

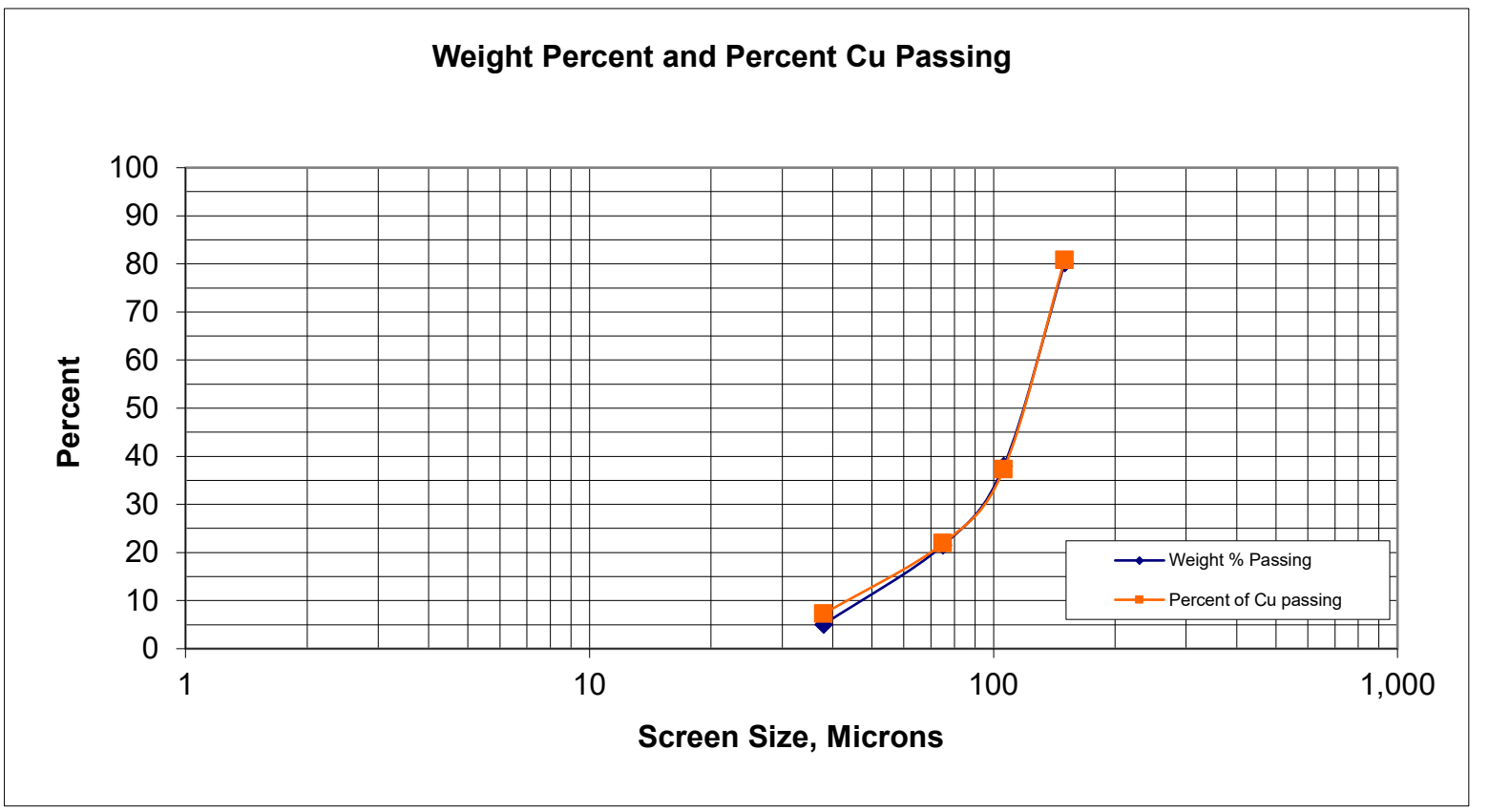
Purpose: To determine the distribution of copper in the sample.

Sample: Approximately 1230 grams of FT 17 Tails (Endo/Exo Master Composite).

Procedure: The sample was screened with screens ranging from 100 mesh to 400 mesh. A split of the minus 10 mesh material was screened
30-minutes (Ro-Tap). The screen fractions were weighed and a representative sample from each fraction was submitted for assa

Results:

Product		Weight						
Fraction (Retained)	microns	grams	Direct %	Cumm % Retained	Analysis Cu (%)	% Dist. Cu	Cumm % Cu Passing	Assay, Cu (%) in screen undersize
Feed (Calculated)		443.3	100.0		0.21	100.0		
+100 mesh	150	88.0	19.9	19.9	0.21	19.2	80.8	0.21
150 mesh x 100 mesh	106	187.0	42.2	62.0	0.22	43.5	37.3	0.21
200 mesh x 150 mesh	75	73.0	16.5	78.5	0.20	15.4	21.9	0.22
400 mesh x 200 mesh	38	73.0	16.5	95.0	0.19	14.7	7.2	0.30
- 400 mesh	-38	22.4	5.0	100.0	0.30	7.2		



Appendix D

Particle Size Analysis 1

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

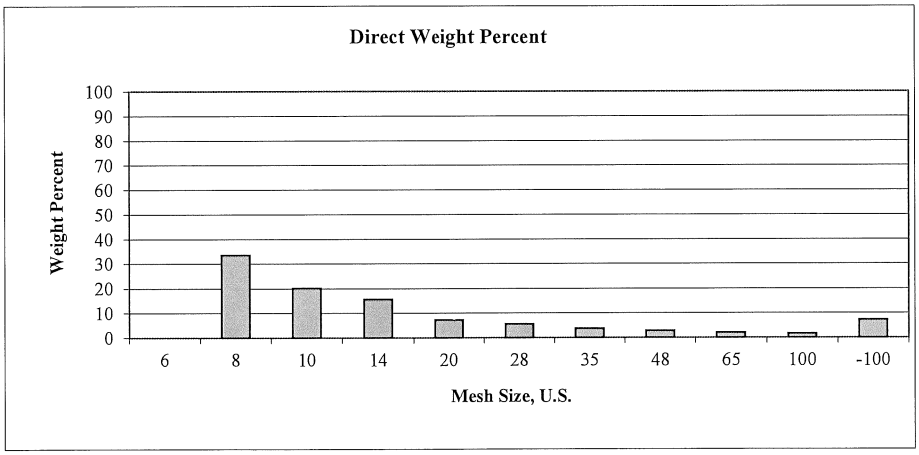
Sample: Approximately 250 grams of FT20 Feed prior to grind (-6 mesh).

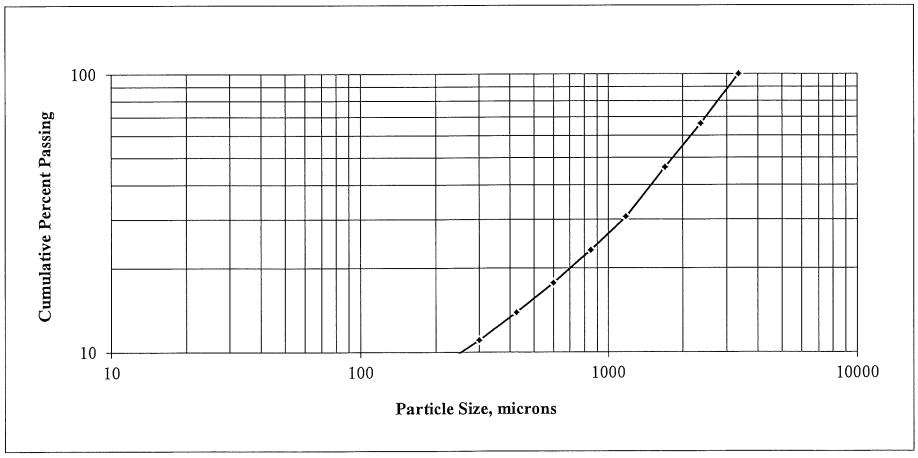
Procedure: The sample was dry screened (Ro-Tap) for 30-minutes on the screens shown. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		245.7	100.0		
6	3350	0.0	0.00	100.00	0.00
8	2360	82.7	33.66	66.34	33.66
10	1700	49.6	20.19	46.15	53.85
14	1180	38.2	15.55	30.61	69.39
20	850	18.1	7.37	23.24	76.76
28	600	13.6	5.54	17.70	82.30
35	425	9.4	3.83	13.88	86.12
48	300	7.0	2.85	11.03	88.97
65	212	5.0	2.04	8.99	91.01
100	150	4.1	1.67	7.33	92.67
-100	-150	18.0	7.33	0.00	100.00

Product Size Passing 80% (P80): = 2,762 Microns





Appendix D

Particle Size Analysis 2

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

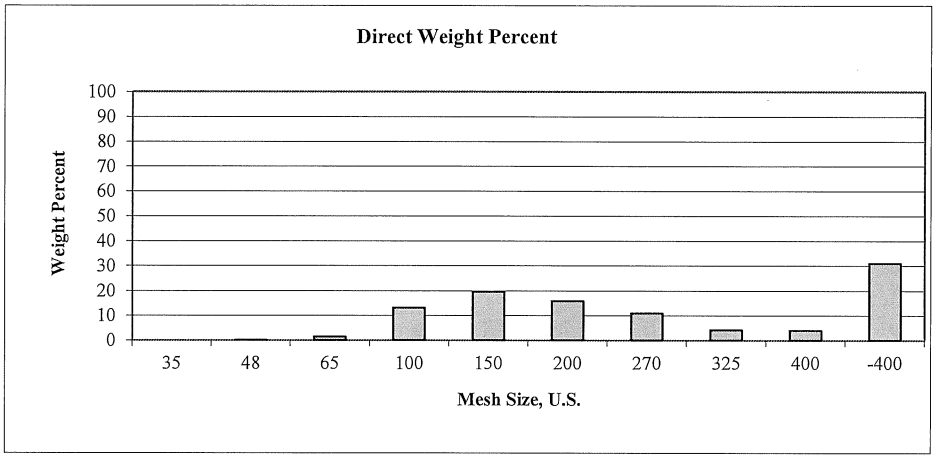
Sample: Approximately 250 grams of FT20 Tails.

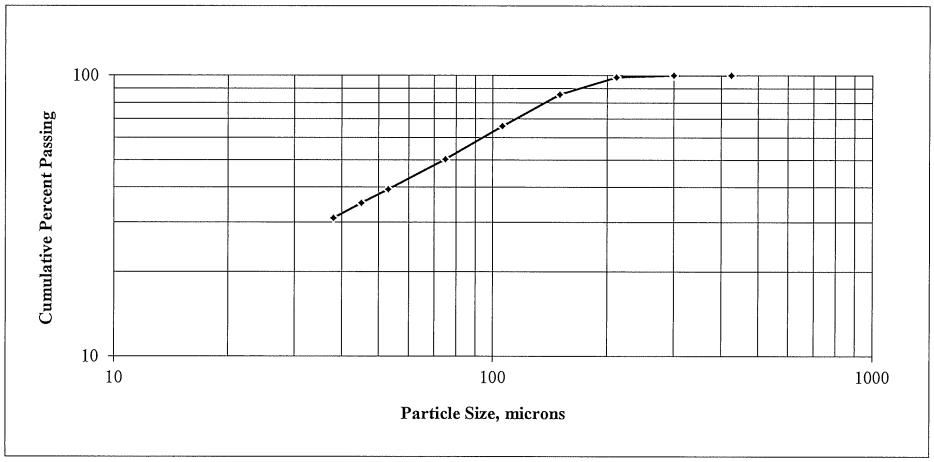
Procedure: The sample was wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		249.2	100.0		
35	425	0.0	0.00	100.00	0.00
48	300	0.1	0.04	99.96	0.04
65	212	3.4	1.36	98.60	1.40
100	150	32.5	13.04	85.55	14.45
150	106	48.6	19.50	66.05	33.95
200	75	39.4	15.81	50.24	49.76
270	53	27.3	10.96	39.29	60.71
325	45	10.6	4.25	35.03	64.97
400	38	10.0	4.01	31.02	68.98
-400	-38	77.3	31.02	0.00	100.00

Product Size Passing 80% (P80): = 137 Microns





Appendix D

Project: 23009 Empire
Date: 13-Sep-23

Particle Size Analysis 3

Purpose: To determine the particle size distribution of the sample.

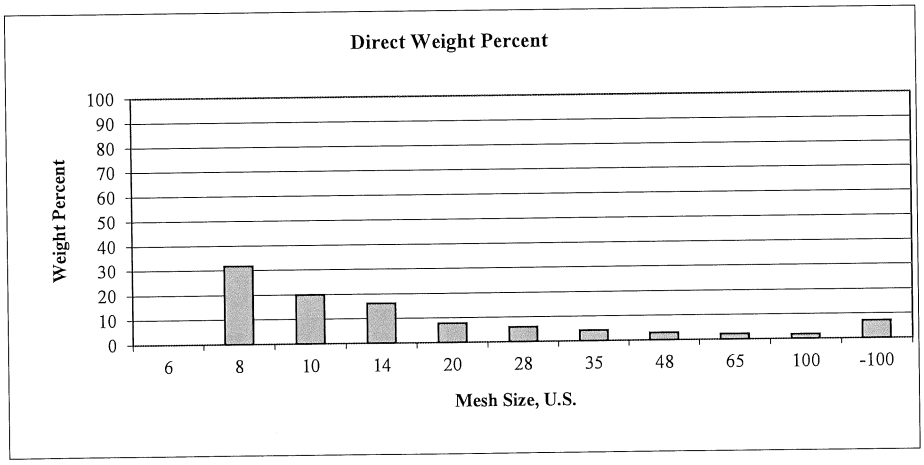
Sample: Approximately 250 grams of FT21 Feed prior to grind (-6 mesh).

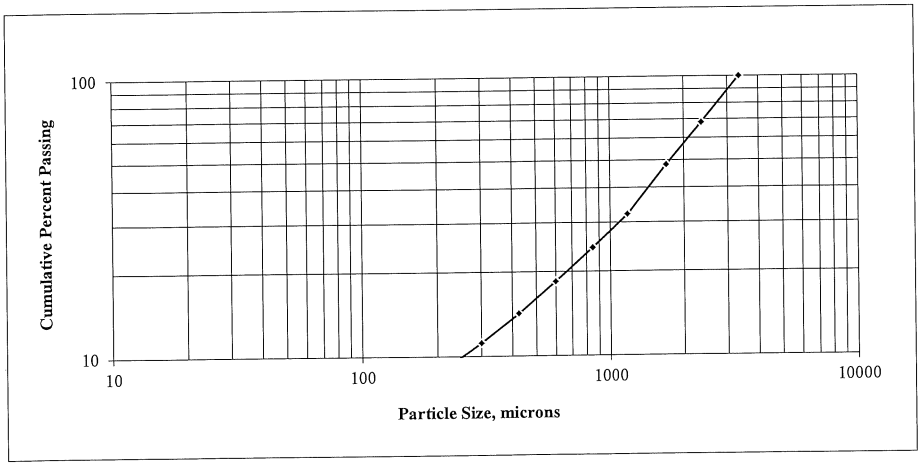
Procedure: The sample was dry screened (Ro-Tap) for 30-minutes on the screens shown. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		245.9	100.0		
6	3350	0.0	0.00	100.00	0.00
8	2360	78.1	31.76	68.24	31.76
10	1700	49.2	20.01	48.23	51.77
14	1180	39.7	16.14	32.09	67.91
20	850	19.0	7.73	24.36	75.64
28	600	14.6	5.94	18.42	81.58
35	425	10.4	4.23	14.19	85.81
48	300	7.5	3.05	11.14	88.86
65	212	5.3	2.16	8.99	91.01
100	150	4.4	1.79	7.20	92.80
-100	-150	17.7	7.20	0.00	100.00

Product Size Passing 80% (P80): = 2,727 Microns





Appendix D

Particle Size Analysis 4

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

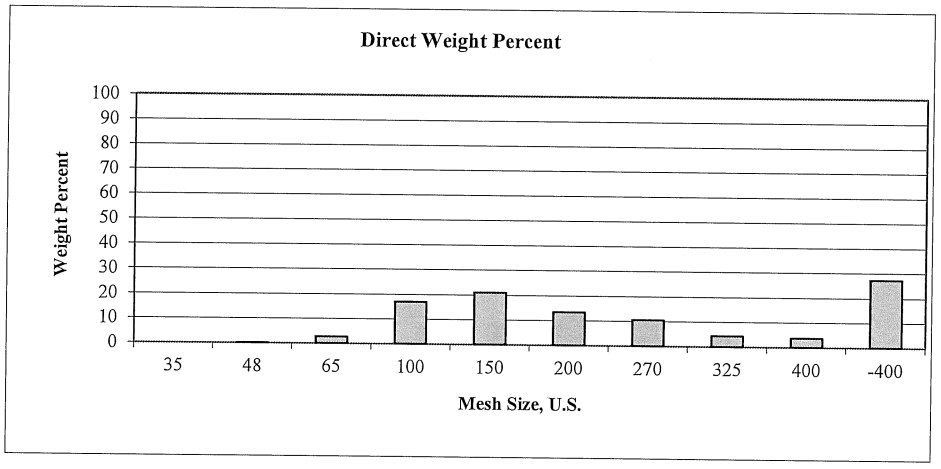
Sample: Approximately 250 grams of FT21 Tails.

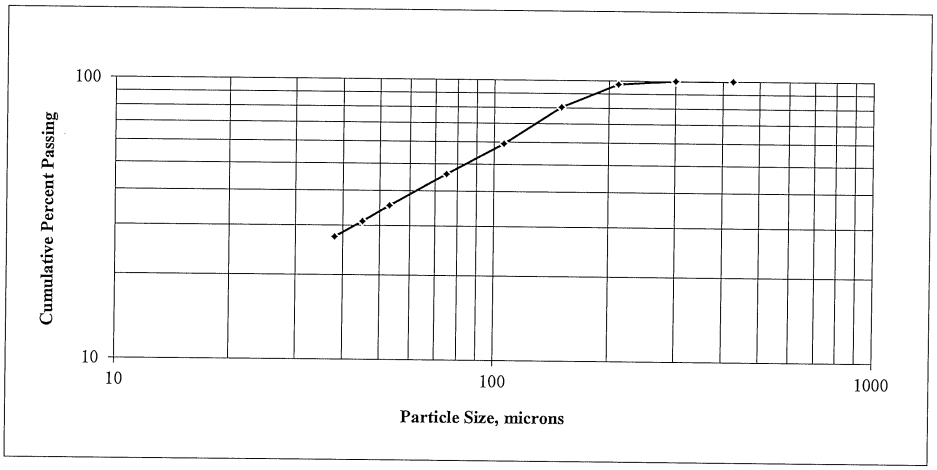
Procedure: The sample was wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		246.5	100.0		
35	425	0.0	0.00	100.00	0.00
48	300	0.3	0.12	99.88	0.12
65	212	6.5	2.64	97.24	2.76
100	150	41.9	17.00	80.24	19.76
150	106	51.7	20.97	59.27	40.73
200	75	32.8	13.31	45.96	54.04
270	53	26.1	10.59	35.38	64.62
325	45	10.7	4.34	31.03	68.97
400	38	9.2	3.73	27.30	72.70
-400	-38	67.3	27.30	0.00	100.00

Product Size Passing 80% (P80): = 149 Microns





Appendix D

Project: 23009 Empire
Date: 13-Sep-23

Particle Size Analysis 5

Purpose: To determine the particle size distribution of the sample.

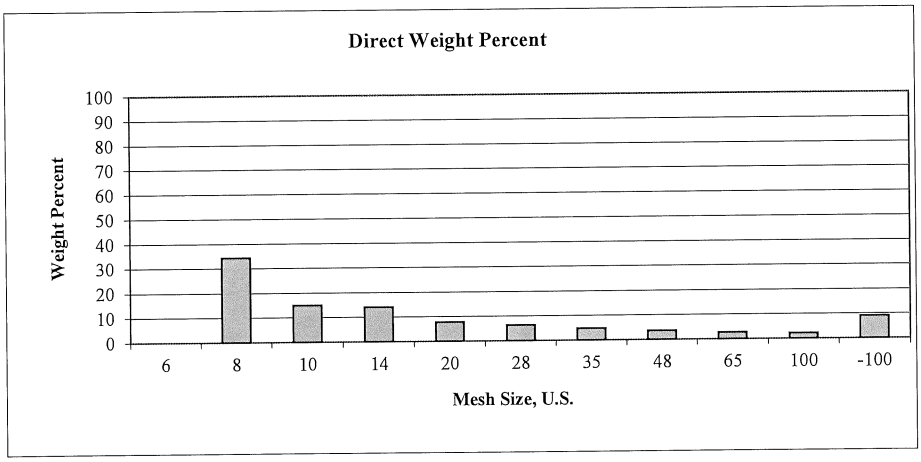
Sample: Approximately 250 grams of FT22 Feed prior to grind (-6 mesh).

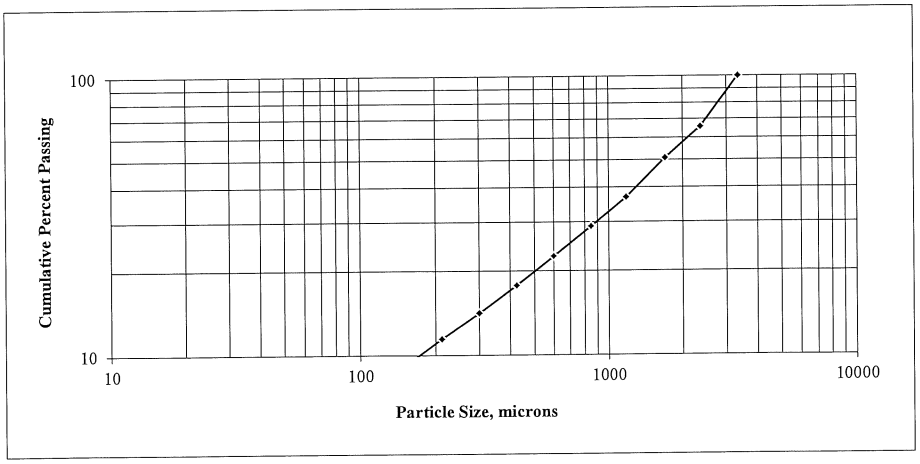
Procedure: The sample was dry screened (Ro-Tap) for 30-minutes on the screens shown. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		240.9	100.0		
6	3350	0.0	0.00	100.00	0.00
8	2360	82.6	34.29	65.71	34.29
10	1700	35.9	14.90	50.81	49.19
14	1180	33.8	14.03	36.78	63.22
20	850	18.9	7.85	28.93	71.07
28	600	15.3	6.35	22.58	77.42
35	425	11.6	4.82	17.77	82.23
48	300	8.8	3.65	14.11	85.89
65	212	6.5	2.70	11.42	88.58
100	150	5.5	2.28	9.13	90.87
-100	-150	22.0	9.13	0.00	100.00

Product Size Passing 80% (P80): = 2,773 Microns





Appendix D

Particle Size Analysis 6

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

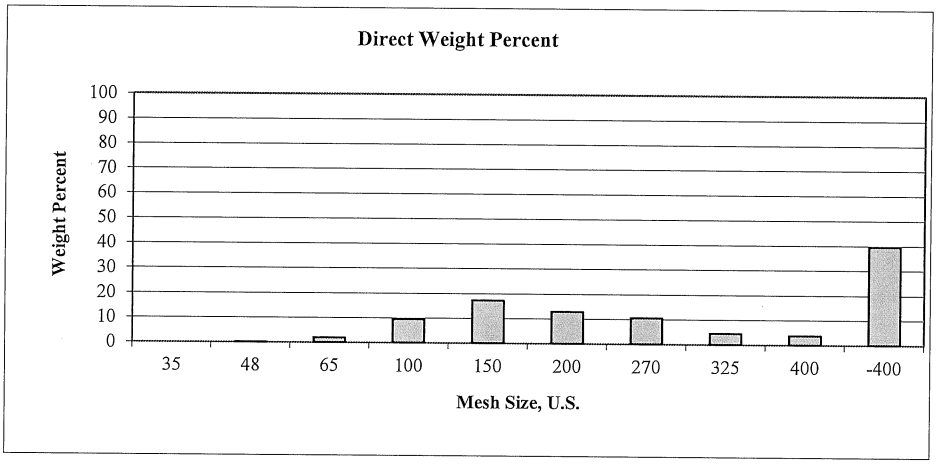
Sample: Approximately 250 grams of FT22 Tails.

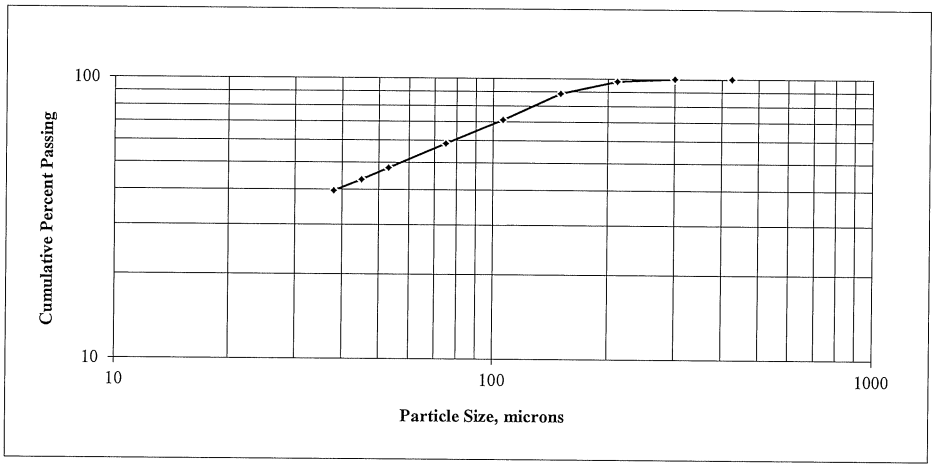
Procedure: The sample was wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		229.8	100.0		
35	425	0.0	0.00	100.00	0.00
48	300	0.3	0.13	99.87	0.13
65	212	4.4	1.91	97.95	2.05
100	150	21.8	9.49	88.47	11.53
150	106	39.6	17.23	71.24	28.76
200	75	29.4	12.79	58.44	41.56
270	53	24.4	10.62	47.82	52.18
325	45	10.3	4.48	43.34	56.66
400	38	8.6	3.74	39.60	60.40
-400	-38	91.0	39.60	0.00	100.00

Product Size Passing 80% (P80): = 128 Microns





Appendix D

Project: 23009 Empire
Date: 13-Sep-23

Particle Size Analysis 7

Purpose: To determine the particle size distribution of the sample.

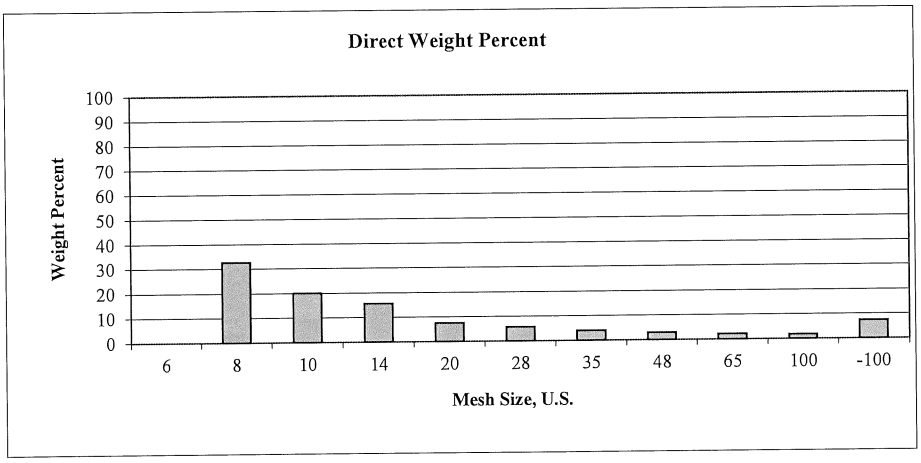
Sample: Approximately 250 grams of FT23 Feed prior to grind (-6 mesh).

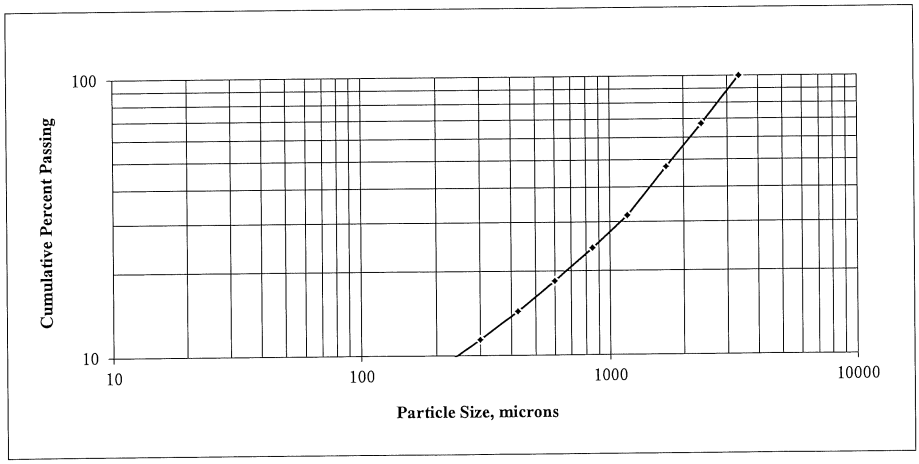
Procedure: The sample was dry screened (Ro-Tap) for 30-minutes on the screens shown. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns			Cumulative	
		Grams	%	Passing	Retained
Feed (Calculated)		240.9	100.0		
6	3350	0.0	0.00	100.00	0.00
8	2360	78.7	32.67	67.33	32.67
10	1700	48.4	20.09	47.24	52.76
14	1180	37.7	15.65	31.59	68.41
20	850	18.0	7.47	24.12	75.88
28	600	13.7	5.69	18.43	81.57
35	425	9.8	4.07	14.36	85.64
48	300	7.2	2.99	11.37	88.63
65	212	5.2	2.16	9.22	90.78
100	150	4.4	1.83	7.39	92.61
-100	-150	17.8	7.39	0.00	100.00

Product Size Passing 80% (P80): = 2,744 Microns





Appendix D

Particle Size Analysis 8

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

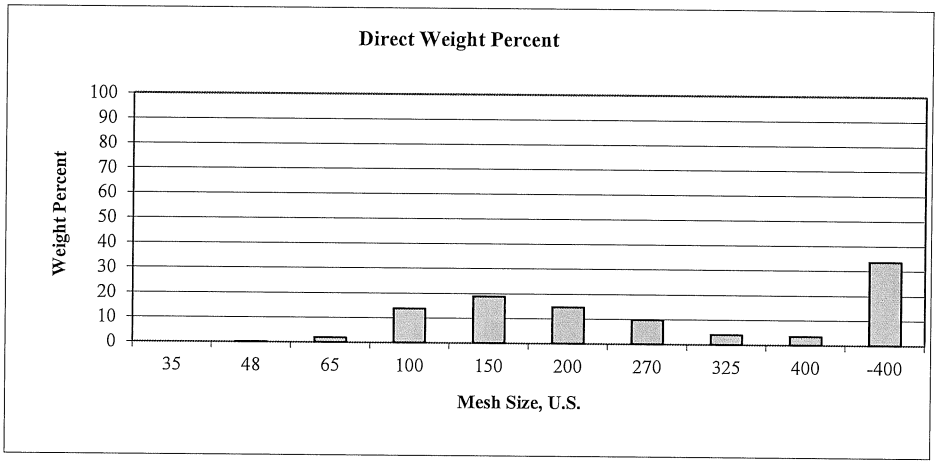
Sample: Approximately 250 grams of FT23 Tails.

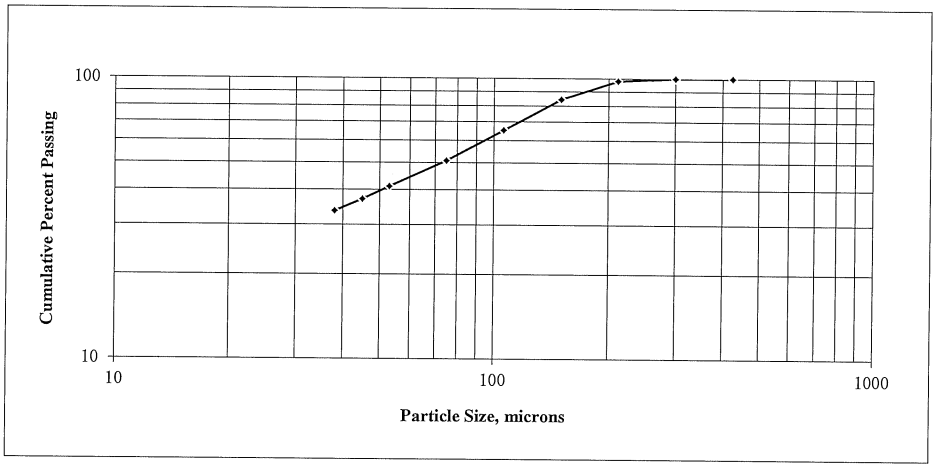
Procedure: The sample was wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		241.0	100.0		
35	425	0.0	0.00	100.00	0.00
48	300	0.2	0.08	99.92	0.08
65	212	4.7	1.95	97.97	2.03
100	150	33.2	13.78	84.19	15.81
150	106	45.3	18.80	65.39	34.61
200	75	35.3	14.65	50.75	49.25
270	53	23.4	9.71	41.04	58.96
325	45	9.8	4.07	36.97	63.03
400	38	8.2	3.40	33.57	66.43
-400	-38	80.9	33.57	0.00	100.00

Product Size Passing 80% (P80): = 140 Microns





Appendix D

Project: 23009 Empire
Date: 13-Sep-23

Particle Size Analysis 9

Purpose: To determine the particle size distribution of the sample.

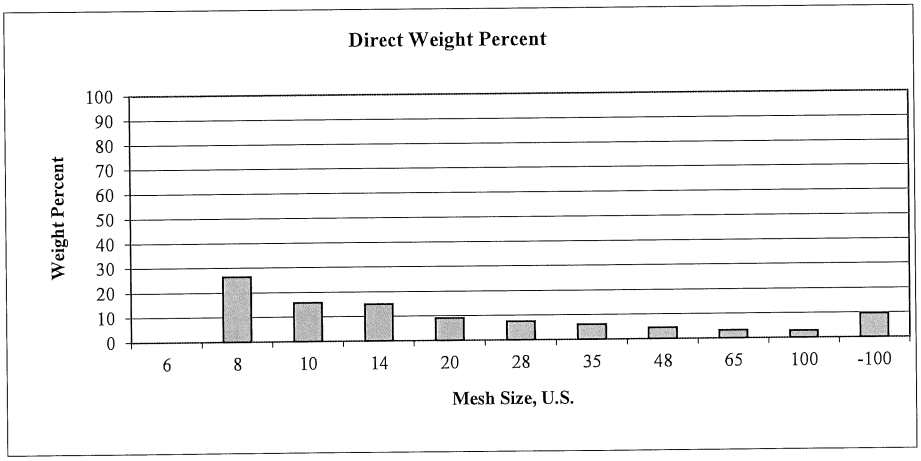
Sample: Approximately 250 grams of FT24 Feed prior to grind (-6 mesh).

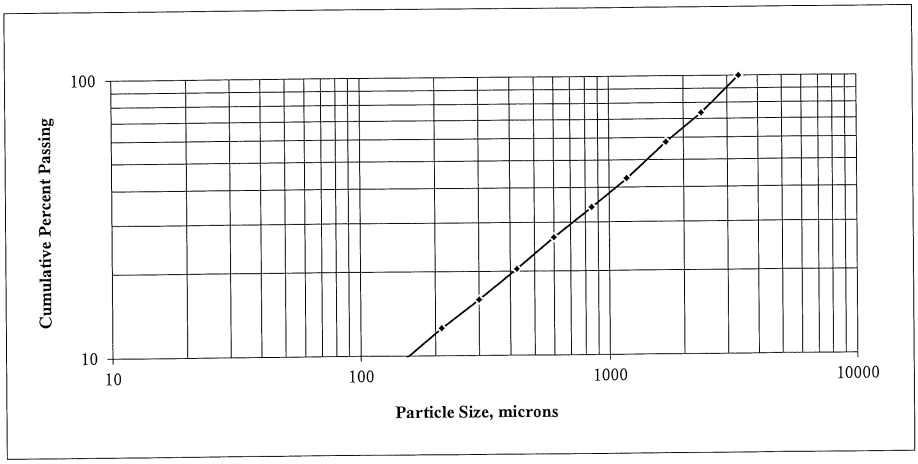
Procedure: The sample was dry screened (Ro-Tap) for 30-minutes on the screens shown. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		241.1	100.0		
6	3350	0.0	0.00	100.00	0.00
8	2360	63.7	26.42	73.58	26.42
10	1700	37.9	15.72	57.86	42.14
14	1180	36.0	14.93	42.93	57.07
20	850	21.8	9.04	33.89	66.11
28	600	18.0	7.47	26.42	73.58
35	425	14.5	6.01	20.41	79.59
48	300	11.0	4.56	15.84	84.16
65	212	8.0	3.32	12.53	87.47
100	150	6.9	2.86	9.66	90.34
-100	-150	23.3	9.66	0.00	100.00

Product Size Passing 80% (P80): = 2,601 Microns





Appendix D

Particle Size Analysis 10

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

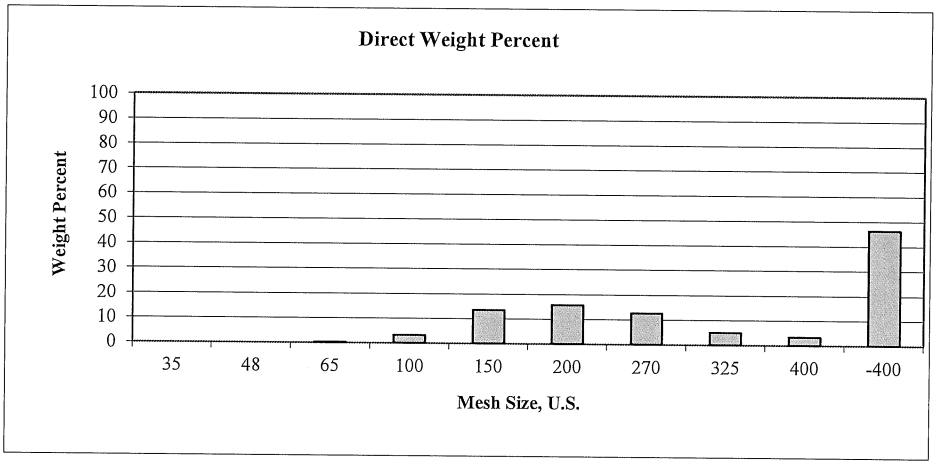
Sample: Approximately 250 grams of FT24 Tails.

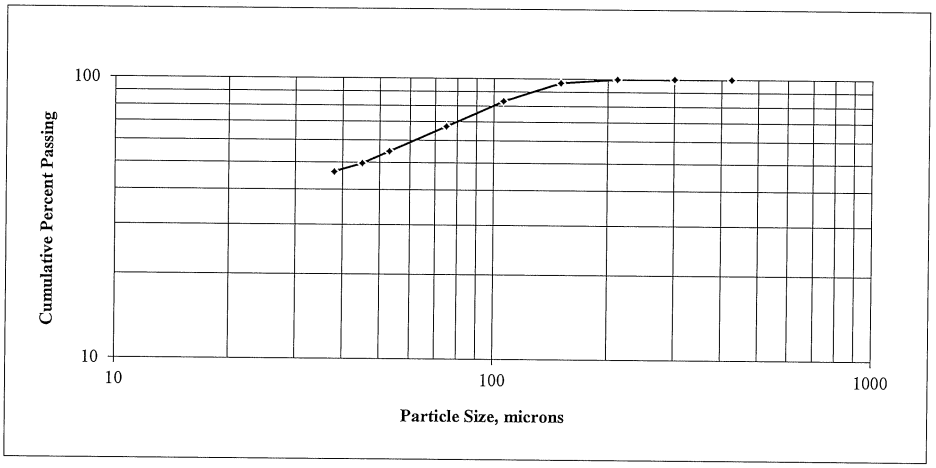
Procedure: The sample was wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		240.7	100.0		
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	0.6	0.25	99.75	0.25
100	150	7.6	3.16	96.59	3.41
150	106	32.5	13.50	83.09	16.91
200	75	37.8	15.70	67.39	32.61
270	53	30.5	12.67	54.72	45.28
325	45	12.2	5.07	49.65	50.35
400	38	8.1	3.37	46.28	53.72
-400	-38	111.4	46.28	0.00	100.00

Product Size Passing 80% (P80): = 100 Microns





Appendix D

Project: 23009 Empire
Date: 13-Sep-23

Particle Size Analysis 11

Purpose: To determine the particle size distribution of the sample.

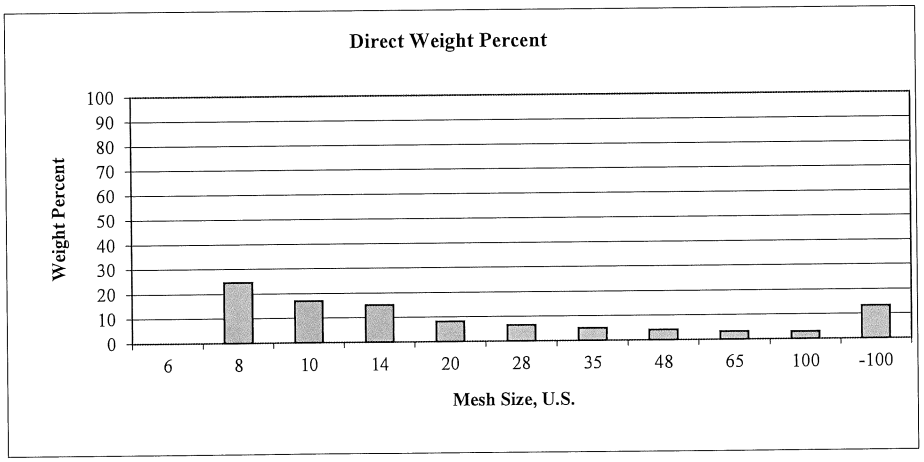
Sample: Approximately 250 grams of FT25 Feed prior to grind (-6 mesh).

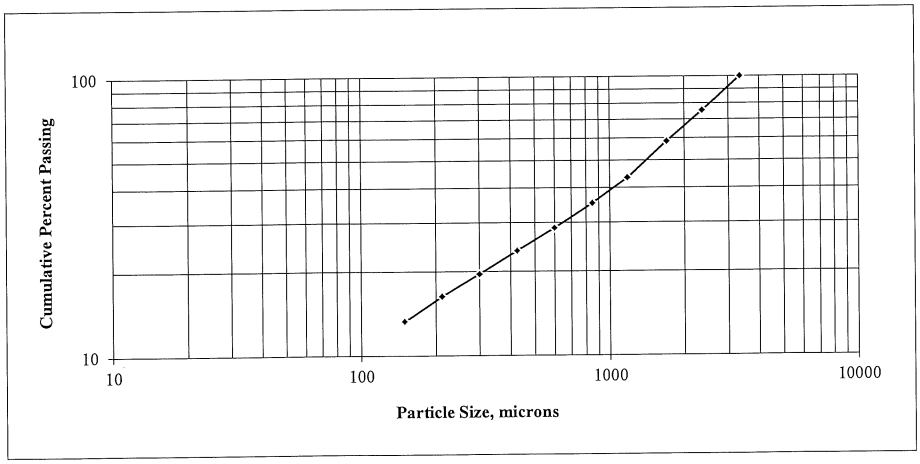
Procedure: The sample was dry screened (Ro-Tap) for 30-minutes on the screens shown. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		233.6	100.0		
6	3350	0.0	0.00	100.00	0.00
8	2360	57.4	24.57	75.43	24.57
10	1700	39.7	16.99	58.43	41.57
14	1180	35.3	15.11	43.32	56.68
20	850	19.1	8.18	35.15	64.85
28	600	15.1	6.46	28.68	71.32
35	425	11.6	4.97	23.72	76.28
48	300	9.6	4.11	19.61	80.39
65	212	7.7	3.30	16.31	83.69
100	150	7.1	3.04	13.27	86.73
-100	-150	31.0	13.27	0.00	100.00

Product Size Passing 80% (P80): = 2,544 Microns





Appendix D

Particle Size Analysis 12

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

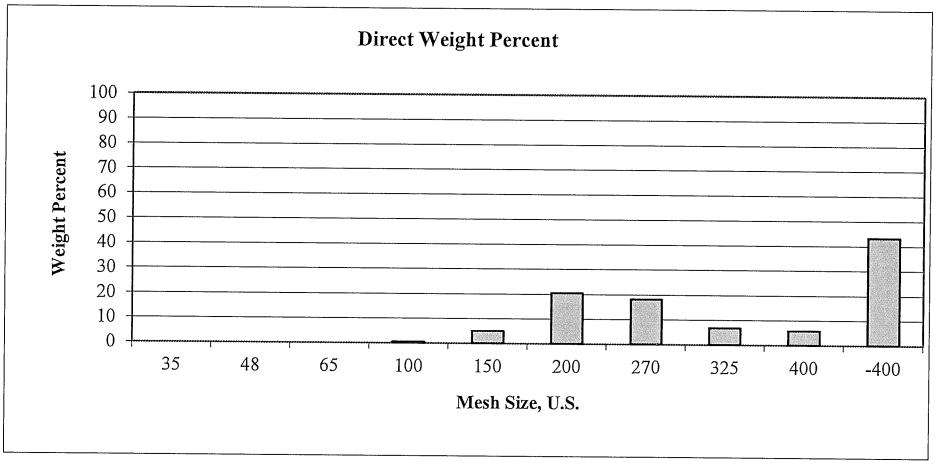
Sample: Approximately 250 grams of FT25 Tails.

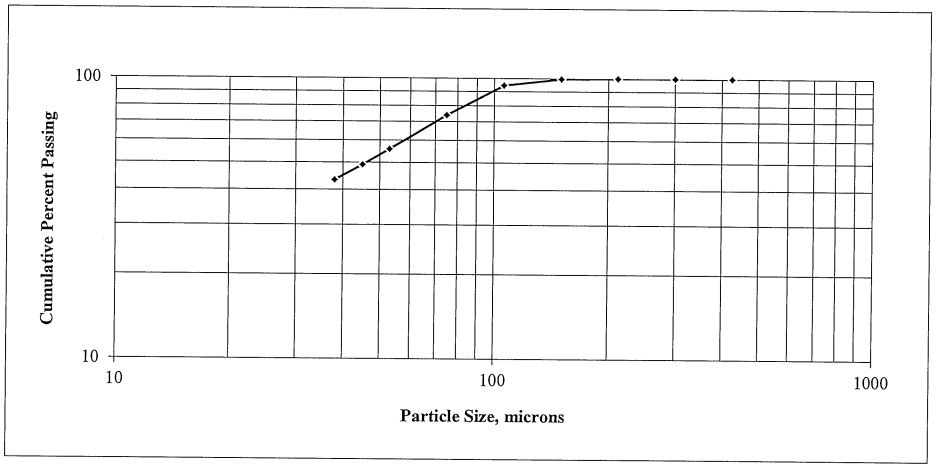
Procedure: The sample was wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		229.4	100.0		
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	0.0	0.00	100.00	0.00
100	150	1.2	0.52	99.48	0.52
150	106	11.6	5.06	94.42	5.58
200	75	46.9	20.44	73.98	26.02
270	53	41.7	18.18	55.80	44.20
325	45	15.6	6.80	49.00	51.00
400	38	13.2	5.75	43.24	56.76
-400	-38	99.2	43.24	0.00	100.00

Product Size Passing 80% (P80): = 84 Microns





Appendix D

Particle Size Analysis 13

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

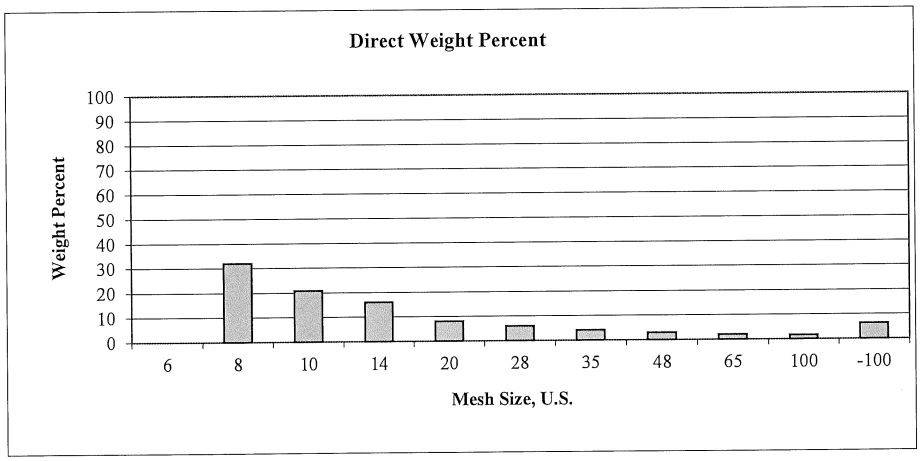
Sample: Approximately 250 grams of FT26 Feed prior to grind (-6 mesh).

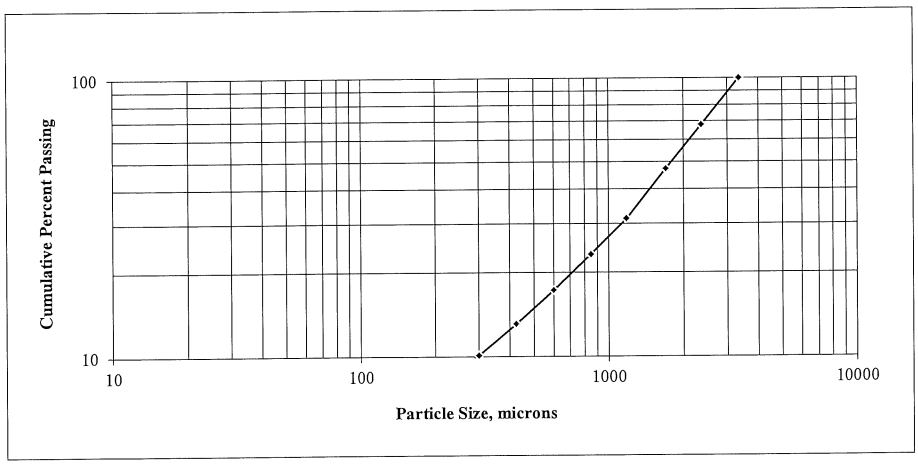
Procedure: The sample was dry screened (Ro-Tap) for 30-minutes on the screens shown. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		227.1	100.0		
6	3350	0.0	0.00	100.00	0.00
8	2360	72.6	31.97	68.03	31.97
10	1700	47.3	20.83	47.20	52.80
14	1180	36.3	15.98	31.22	68.78
20	850	18.2	8.01	23.21	76.79
28	600	13.5	5.94	17.26	82.74
35	425	9.5	4.18	13.08	86.92
48	300	6.8	2.99	10.08	89.92
65	212	4.8	2.11	7.97	92.03
100	150	3.8	1.67	6.30	93.70
-100	-150	14.3	6.30	0.00	100.00

Product Size Passing 80% (P80): = 2,731 Microns





Appendix D

Project: 23009 Empire
Date: 13-Sep-23

Particle Size Analysis 14

Purpose: To determine the particle size distribution of the sample.

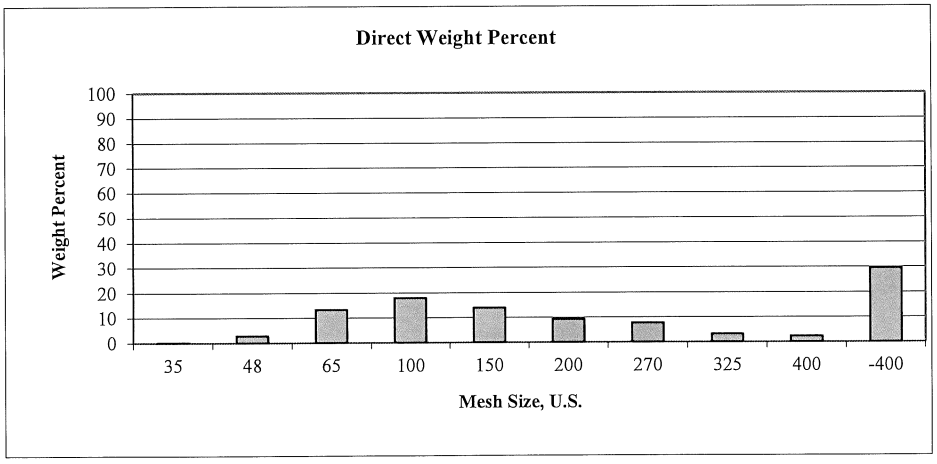
Sample: Approximately 250 grams of FT260 Tails.

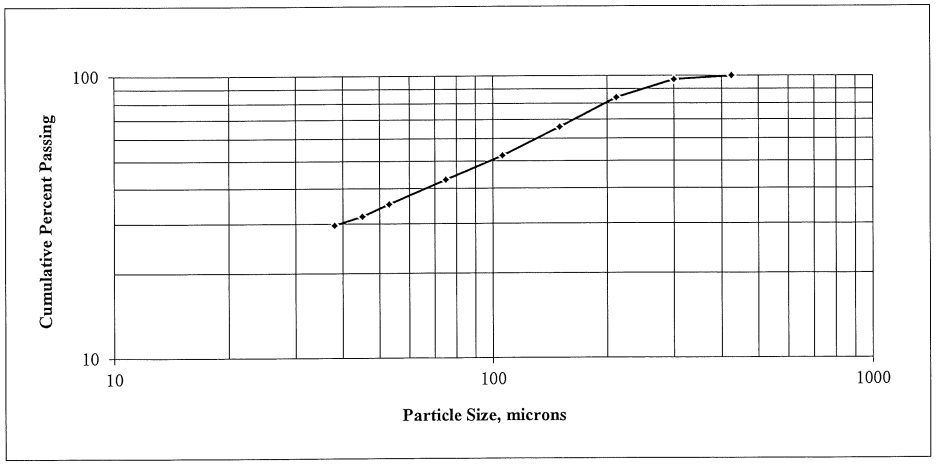
Procedure: The sample was wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		238.6	100.0		
35	425	0.2	0.08	99.92	0.08
48	300	6.8	2.85	97.07	2.93
65	212	31.7	13.29	83.78	16.22
100	150	42.4	17.77	66.01	33.99
150	106	33.0	13.83	52.18	47.82
200	75	22.2	9.30	42.88	57.12
270	53	18.4	7.71	35.16	64.84
325	45	7.8	3.27	31.89	68.11
400	38	5.5	2.31	29.59	70.41
-400	-38	70.6	29.59	0.00	100.00

Product Size Passing 80% (P80): = 199 Microns





Appendix D

Particle Size Analysis 15

Project: 23009 Empire
Date: 13-Sep-23

Purpose: To determine the particle size distribution of the sample.

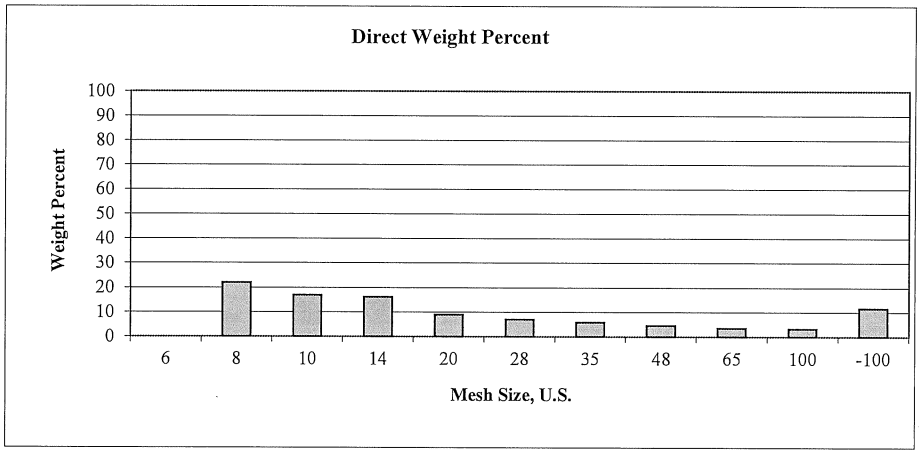
Sample: Approximately 250 grams of FT27 Feed prior to grind (-6 mesh).

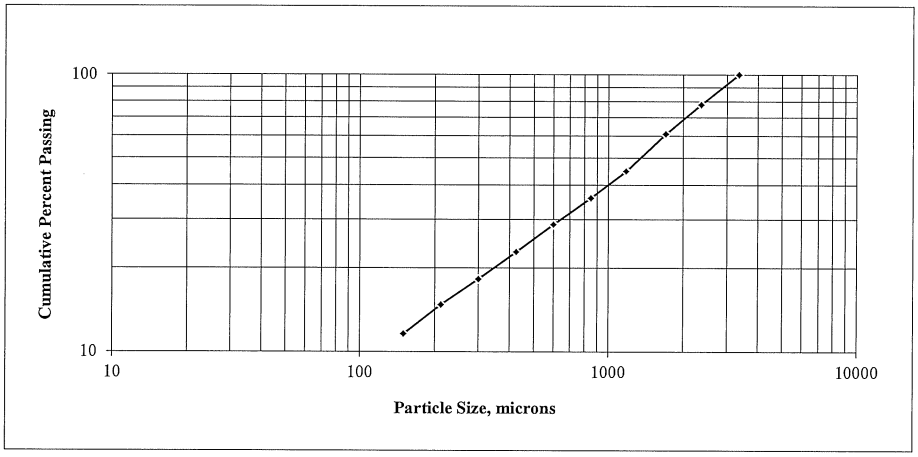
Procedure: The sample was dry screened (Ro-Tap) for 30-minutes on the screens shown. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		524.0	100.0		
6	3350	0.0	0.00	100.00	0.00
8	2360	115.5	22.04	77.96	22.04
10	1700	88.9	16.97	60.99	39.01
14	1180	84.9	16.20	44.79	55.21
20	850	47.1	8.99	35.80	64.20
28	600	37.2	7.10	28.70	71.30
35	425	30.7	5.86	22.84	77.16
48	300	24.1	4.60	18.24	81.76
65	212	18.3	3.49	14.75	85.25
100	150	16.6	3.17	11.58	88.42
-100	-150	60.7	11.58	0.00	100.00

Product Size Passing 80% (P80): = 2,452 Microns





www.fortedynamics.com
120 Commerce Drive, Unit 3-4, Fort Collins, CO 80524
Phone: +1 (720) 642-9359 info@fortedynamics.com