TETRA TECH EBA

PRELIMINARY ECONOMIC ASSESSMENT FOR THE BREWERY CREEK PROPERTY YUKON TERRITORY, CANADA

NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

PRESENTED TO

GOLDEN PREDATOR EXPLORATION LTD.

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REPORT AUTHORS:

Mark Horan, MSc, P.Eng

James Barr, P.Geo.

Marvin Silva, PhD, P.Eng.

Nick Michael, MBA, SME-RM

Donald E. Hulse, P.E., SME-RM

M. Claiborne Newton, PhD, SME-RM

Joseph M. Keane, P.E.

Michael Lechner, C.P.G., P.Geo

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TABLE OF CONTENTS

1.0 EXECUTIVE SUMMARY 1
1.1 Introduction 1
1.2 Property Description and Location 1
1.3 Property History 2
1.4 Geological Setting and Mineralization 2
1.5 Exploration 3
1.6 Mineral Resources 3
1.7 Mining 4
1.8 Electrical Power 5
1.9 Mineral Processing and Metallurgical Test Work 5
1.10 Process Plant 7
1.11 Capital Cost Estimates 7
1.12 Operating Costs 8
1.13 Economics 9
1.14 Interpretation and Conclusions 10
1.14.1 Project Risks 10
1.14.2 Project Opportunities 10
1.15 Recommendations 11
2.0 INTRODUCTION AND TERMS OF REFERENCE 12
2.1 Report Authors and Quality Control 12
2.2 Site Visits 13
2.3 Effective Date of Technical Report 13
3.0 RELIANCE ON OTHER EXPERTS 13
4.0 PROPERTY DESCRIPTION AND LOCATION 13
4.1 Locality Plan/Map 14
4.2 Mineral Titles 14
4.3 Royalties and Agreements 20
4.3.1 Property Purchase Agreement with Alexco Resource Corp. 20
4.3.2 Royalties 21
4.3.2.1 Alexco Royalty 21
4.3.2.2 Franco-Nevada Royalty 21
4.3.2.3 Energold Royalty 21
4.3.2.4 Till Capital Royalty 22
4.3.2.5 Permits and Status 22
5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 22
5.1 Infrastructure 23
5.1.1 Existing Roads 23
5.1.2 Existing Laydown Area 24
5.1.3 Existing Process Ponds 24
5.1.4 Existing Heap Leach Facility 24

6.0 HISTORY OF THE PROPERTY ... 25
6.1 Ownership ... 25
6.2 Exploration ... 26
6.2.1 Geologic Mapping ... 26
6.2.2 Soil Sampling Surveys ... 27
6.2.3 Geophysics ... 27
6.2.4 Trenching ... 28
6.2.5 Drilling ... 28
6.2.6 Norex (1989) ... 28
6.2.7 Loki / Viceroy (1990 – 1999) ... 29
6.2.8 Spectrum (2004) ... 29
6.2.9 Alexco (2006) ... 29
6.3 Production ... 31
7.0 GEOLOGICAL SETTING AND MINERALIZATION ... 31
7.1 Regional Geology ... 31
7.2 Local Geology ... 34
7.2.1 Stratigraphy ... 36
7.2.2 Intrusive Rocks ... 37
7.2.3 Structural Geology ... 38
7.3 Mineralization ... 39
7.3.1 Areas Included in PEA ... 40
7.3.2 Areas Not Included in PEA ... 42
7.4 Local Surficial Geology, Terrain and Permafrost ... 44
7.4.1 Terrain Hazards And Permafrost ... 45
8.0 DEPOSIT TYPES ... 46
8.1 Mineral Deposit ... 46
8.2 Geological Model Applied ... 46
9.0 EXPLORATION ... 47
9.1 Recent Exploration Work ... 47
9.1.1 Airborne Magnetic Surveys, 2011-2012 ... 48
9.1.2 Ground Magnetic Survey, 2012 ... 49
9.1.3 Induced Polarization Survey, 2011 ... 49
9.1.4 Soil Sampling Survey ... 50
10.0 DRILLING ... 50
10.1 Golden Predator Diamond Drilling, 2009 ... 52
10.2 Golden Predator RC Drilling, 2010 ... 52
10.3 Golden Predator Diamond Drilling, 2010 ... 52
10.4 Golden Predator RC Drilling, 2011 ... 52
10.5 Golden Predator Diamond Drilling, 2011 ... 53
10.6 Golden Predator Sonic Drilling, 2011 ... 53
10.7 Golden Predator Diamond Drilling, 2012 ... 53
10.7.1 Golden Predator Metallurgical Drilling, 2012 ... 53
10.7.2 Golden Predator Geotechnical Drilling, 2012 ... 53
10.7.3 Golden Predator Groundwater Monitoring Drilling, 2012 ... 54

10.8 Golden Predator RC Exploration Drilling, 2012 54
10.9 Summary of All Drilling Data 55
11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY 56
11.1 Sample Collection Methods 56
11.1.1 Historical Sampling by Norex, 1989 56
11.1.2 Historical Sampling by Loki and Viceroy, 1990-1999 56
11.1.3 Historical Sampling by Spectrum and Alexco, 2004-2006 57
11.1.4 Golden Predator Core Sampling, 2009-2012 57
11.1.5 Golden Predator RC Sampling, 2010-2011 58
11.1.6 Golden Predator RC Sampling, 2012 58
11.2 Sample Processing and Security Measures 58
11.3 Sample Analytical Methods 61
11.3.1 Historical Analytical Methods by Norex, 1989 61
11.3.2 Historical Analytical Methods by Loki, and Viceroy, 1990 – 1999 61
11.3.3 Historical Analytical Methods by Spectrum, 2004 61
11.3.4 Historical Analytical Methods by Alexco, 2006 61
11.3.5 Golden Predator's Analytical Methods, 2009 61
11.3.6 Golden Predator's Analytical Methods, 2010 61
11.3.7 Golden Predator's Analytical Methods, 2011 62
11.3.8 Golden Predator's Analytical Methods, 2012 62
11.4 Analytical Quality Assurance and Quality Control 62
11.4.1 RMI Review of Database 62
11.4.1.1 2012 Blank Performance 63
11.4.1.2 2012 SRM Performance 64
11.4.1.3 2012 Duplicate Sample Performance 66
11.4.2 Gustavson Review of Database 67
11.4.2.1 Standards 67
11.4.2.2 Blanks 68
11.4.2.3 Duplicates 70
11.5 QAQC Compliance Statements 72
11.5.1 RMI Statement 72
11.5.2 Gustavson Statement 72
11.5.3 Tetra Tech EBA Statement 72
12.0 DATA VERIFICATION 72
12.1 Tetra Tech EBA Verification of Historical Data 72
12.1.1 Tetra Tech EBA Comparison Between Core and RC Data 75
12.1.2 Tetra Tech EBA Comparison Between Recent Core and Historical Core 76
12.1.3 Tetra Tech EBA Statement 78
12.2 Tetra Tech EBA Verification of Recent Data 78
12.2.1 Tetra Tech EBA Discussion of Independent Sample Results 79
12.2.2 Tetra Tech EBA Statement on Recent Data Verification 80
12.3 RMI Data Verification 80
12.3.1 RMI Comparison of Core Versus RC Drilling 81
12.3.2 Drilling-Sampling-Recovery Factors 85

12.3.3 RMI Statement Regarding Data Verification 86
12.4 Data Verification by Gustavson 87
12.4.1 Verification of Historical Data 87
12.4.2 Verification of Drill Data 87
12.4.3 Gustavson Site Visit 87
12.4.4 Gustavson's Statement on Data Adequacy 88

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING 88
13.1 MLI Job No. 3719 Metallurgical Test Work 89
13.2 Head Assay Analyses 89
13.2.1 Screen Analyses 90
13.2.2 Bottle Roll Testing 91
13.2.3 Locked Cycle Column Leach Testing 93
13.2.4 Prediction of Gold Production 99
13.2.5 Prediction of Reagent Consumption 101
13.3 MLI Job No. 3618 Reprocessing of Original Heap 103
13.3.1 Reprocessed Ore Sample Preparation 103
13.3.2 Reprocessed Ore Column Leach Tests 104
13.4 Gold Extraction and Reagent Consumption 105
13.5 Crushing Work Index and Abrasion 106

14.0 MINERAL RESOURCE ESTIMATES 107
14.1 Kokanee, Golden, Pacific, Blue, and Lucky Deposits 109
14.1.1 Deposit Geology Pertinent to Resource Estimation 109
14.1.2 Data Used for Estimation 109
14.1.3 Bulk Density 110
14.1.4 Methodology 110
14.1.5 Capping of Assays 111
14.1.6 Compositing 112
14.1.7 Variography 112
14.1.8 Estimation 113
14.1.8.1 Estimate Validation 114
14.1.9 Review of Estimate – Gustavson 120
14.2 Bohemian, Schooner, Lower Fosters, West Big Rock, East Big Rock, Classic, and Lone Star Deposits 120
14.2.1 Deposit Geology Pertinent to Resource Estimation 121
14.2.1.1 Oxidation 121
14.2.2 Data Used for Estimation 123
14.2.2.1 Drillhole Assay Statistics 123
14.2.2.2 Topographic Data 128
14.2.3 Bulk Density 128
14.2.4 Methodology 129
14.2.5 Capping of Assays 129
14.2.6 Compositing 130
14.2.7 Variography 131
14.2.8 Estimation 135

14.2.8.1 Model Validation 136
14.3 Historical Heap Leach Pad 145
14.4 North Slope and Sleeman Deposits 147
14.4.1 Deposit Geology Pertinent to Resource Estimation 148
14.4.1.1 Oxidation 148
14.4.2 Data Used for Estimation 149
14.4.3 Bulk Density 149
14.4.4 Methodology 150
14.4.5 Capping of Assays 151
14.4.6 Compositing 151
14.4.7 Estimation 152
14.4.7.1 Grade Cut-off Selection 154
14.4.7.2 Model Validation 156
14.4.7.3 Visual Comparison 156
14.4.7.4 Geostatistical Comparison 156
14.4.7.5 Nearest Neighbor Comparison 157
14.5 Mineral Resource Classification 158
14.5.1 Kokanee, Golden, Pacific, Blue, and Lucky Deposits 159
14.5.2 Bohemian, Schooner, Lower Fosters, West Big Rock, East Big Rock, Classic, and Lone Star Deposits 159
14.5.3 Historical Heap Leach Pad 159
14.5.4 North Slope and Sleeman Deposits 159
14.6 Mineral Resource Estimation 160
15.0 MINERAL RESERVE ESTIMATES 162
16.0 MINING METHODS 162
16.1 Introduction 162
16.2 Lerch-Grossman Optimization 163
16.3 Pit Slope Design 163
16.3.1 Rock Mass Classification 164
16.3.2 Geotechnical and Structural Domains 164
16.3.3 Rock Mass Structures 164
16.3.4 Rock Slope Stability Assessment 165
16.3.5 Pit Slope Assessment – Kinematic Analysis 165
16.3.6 Rock Mass Strength Analysis 167
16.3.7 Recommended Slope Configuration 168
16.3.7.1 Bench Face Angle and Inter-ramp Angle Recommendations 168
16.3.7.2 Bench Width Recommendations 168
16.3.8 Open Pit Hydrology 168
16.4 Haul Roads 169
16.5 Design of the Open Pits 172
16.5.1 Pit Designs Results 173
16.5.2 Mine Plan and Mine Schedule 182
16.5.3 Scheduling Results 182
16.6 Waste Rock Storage 184

16.7 Geometry of Potential External Waste Storage Embankments 184
16.7.1 Waste rock storage facility designs 186
16.8 Waste Rock Management 197
16.8.1 Construction of Waste Rock Storage Facilities and Pre-development 197
16.9 Mining Equipment Fleet 197
16.9.1 Haul truck cycle times 198
16.9.1.1 Horizontal Curves 199
16.9.1.2 Vertical Curves: 199
16.9.1.3 Total cycle 199
16.9.2 Drilling and Blasting 200
16.9.3 Loading and Hauling 200
16.9.4 Support equipment 201
16.10 Drainage and Dewatering, and Electrical Services 201
16.11 Mining Equipment, Labour and Consumables Schedule 201

17.0 RECOVERY METHODS 202

17.1 Heap Leaching 202
17.1.1 Engineering Design Criteria 202
17.1.2 Heap Pad Slope Stability 206
17.1.3 Settlement Assessment 207
17.1.4 Heap Pad Surface Water Management 207
17.1.5 Heap Leach Water Balance 207
17.1.6 Heap Leach Liner Design 208
17.1.7 Containment Dyke West Side of Cells 8 to 10 209
17.1.8 Heap Leach Operations 209
17.2 Recovery Methods 210
17.2.1 Process Plant Overview 210
17.2.2 Crushing 213
17.2.3 Heap Leach Stacking and Placement 215
17.2.4 Solution Management 217
17.2.5 Carbon Adsorption 219
17.2.6 Stripping and Refining 221
17.2.7 Acid Washing and Carbon Reactivation 223
17.2.8 Heap Detoxification (Future) 225
17.2.9 Reagents 227
17.2.10 Water 227

18.0 PROJECT INFRASTRUCTURE 227

18.1 Mine Site Infrastructure Required to Restart Brewery Creek mine 228
18.1.1 Planned Infrastructure Overview 228
18.2 Planned Layout 229
18.3 ROM Storage and Crushing Facility 235
18.3.1 Crushing Plant Site Layout 235
18.4 Process Plant Site 235
18.5 Crushing and Process Plant Site Preparation and Earthworks 235
18.5.1 Crushing and Process Plant Internal Roads 235

18.5.2 Tanks and Miscellaneous Equipment 236
18.6 Temporary Stockpiles for Recovered Heap Leach Material 236
18.7 Concrete Work 236
18.8 Steel 236
18.9 Site Geotechnical 237
18.9.1 Foundations 237
18.9.2 Site Grading 237
18.10 Water Consumption 237
18.10.1 Permissible Water Extraction Points 238
18.10.2 Water Supply 238
18.10.3 ADR Facility Water 238
18.10.4 Truck Shop, Offices and Camp Water 238
18.10.5 Fire Water 238
18.10.6 Potable Water Treatment (Camp) 238
18.11 Waste Water Management 239
18.11.1 Sewage and Grey Water Disposal 239
18.11.2 Truck Shop Wash Bay Water 239
18.11.3 Run-off Treatment 239
18.12 Solid Waste Management 240
18.13 Support Infrastructure 241
18.13.1 Fuel Storage and Distribution 241
18.13.2 Propane 241
18.13.3 Explosives Storage 241
18.14 Buildings 241
18.14.1 Process Plant Building 242
18.14.2 Truck shop 242
18.14.3 Warehouse 242
18.14.4 Mine Dry 242
18.14.5 Assay Lab 243
18.14.6 Substation Buildings 243
18.14.7 Administration Buildings 243
18.14.8 Restrooms 243
18.15 Site and Access Roads 243
18.15.1 Access Roads 243
18.15.2 Mine Site and Haul Roads 243
18.15.3 Geotechnical and Environmental 243
18.16 Power Supply and Distribution 243
18.16.1 Switchgear and Motor Control Centers 244
18.16.2 Instrumentation 245
18.16.3 Site Wide Distribution 245
18.16.4 Grid Power Supply Trade-Off 246
18.16.4.1 Tap Point Substation 246
18.16.4.2 Transmission/Distribution Line 246
18.16.4.3 Brewery Creek Mine Substation 247
18.16.5 LNG Power Supply Study 247

18.17 Lighting and Grounding 247
18.18 Communication 247
18.19 Fire, Life and Emergency Response 247

19.0 MARKET STUDIES AND CONTRACTS 248
19.1 Gold Marketing 248
19.2 Equipment Leasing Contracts 248

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT 248
20.1 Historical Studies 248
20.2 Recent Studies 249
20.2.1 YESAB Submission 249
20.2.1.1 Geochemical Characterization of Waste Rock 249
20.2.2 Site Hydrology and Water Balance 250
20.2.3 Permits Held by Golden Predator 251
20.2.4 Ongoing Environmental Studies 252
20.3 Tr’ondëk Hwëch’in First Nation Socio Economic Accord 252

21.0 CAPITAL AND OPERATING COSTS 252
21.1 Capital Costs 253
21.1.1 Site Infrastructure 254
21.1.2 Open Pit Mining Capital Estimate 254
21.1.2.1 Pre-stripping and haul road construction 255
21.1.2.2 Mining equipment 255
21.1.2.3 Mining infrastructure 256
21.1.3 Site Work (Earthworks) Costs Included in the Capital Cost Estimate 257
21.1.4 Haul Road Cost Estimates 257
21.1.5 Concrete Construction Rates Used in the Capital Cost Estimate 257
21.1.6 Construction Labour Used in the Capital Cost Estimate 258
21.1.7 Structural Steel 258
21.1.8 Water Management 259
21.1.8.1 Fresh Water 259
21.1.8.2 Process Piping Costs 259
21.1.9 Power Supply 259
21.1.9.1 Electrical – Power Distribution and Instrumentation 259
21.1.9.2 Transformers Sizes 259
21.1.9.3 Switchgear and Control 260
21.1.10 Processing Plant Capital 260
21.1.10.1 Capital Cost Basis for Processing Plant 262
21.1.10.2 Direct Costs 262
21.1.10.3 Equipment Costs 264
21.1.10.4 Construction Consumables for Process Plant 264
21.1.10.5 Freight and Equipment Insurance 264
21.1.10.6 Indirect Costs 264
21.1.10.7 Mobile Equipment for Construction 264
21.1.11 Exclusions for Processing 265
21.1.12 Heap Leach Facility Construction Capital Costs 265

21.2 Operating Costs....267
21.2.1 General and Administrative Costs....268
21.2.2 Mining Costs....270
21.2.3 Mining Equipment Leasing....271
21.2.4 Heap Leach Stacking Costs (Mobile Machinery Option)....271
21.2.5 Processing Plant Operating Costs....271

22.0 ECONOMIC ANALYSIS.... 275
22.1 Introduction....275
22.2 Taxes and Royalties....276
22.3 Base Case Sensitivity Analysis....277
22.4 Results of Trade-Off Studies....278
22.5 Discount Cash Flow....279

23.0 ADJACENT PROPERTIES.... 282

24.0 OTHER RELEVANT DATA AND INFORMATION.... 282
24.1 Project Execution Plan....282
24.1.1 Health, Safety, Environmental and Security....282
24.2 Project Scheduling....282

25.0 INTERPRETATION AND CONCLUSIONS.... 284
25.1 General (Tetra Tech EBA)....284
25.1.1 Risks ……….....284
25.1.2 Opportunities....285
25.2 General (SGS)....286
25.2.1 Risks …………....286
25.2.2 Opportunities....286
25.3 Truck Stacking....286
25.3.1 Risks ………………....286
25.3.2 Opportunities....287
25.4 Plant Site....287
25.4.1 Opportunities....287
25.5 Metallurgical Testwork....287
25.5.1 Risks ………....287

26.0 RECOMMENDATIONS.... 287
26.1 Recommendations....287

REFERENCES.... 289

CERTIFICATES OF QUALIFIED PERSONS

LIST OF TABLES IN TEXT

Table 1-1: Summary of Oxide Mineral Resources....3
Table 1-2: Summary of Sulfide Mineral Resources....4
Table 1-3: Total Tonnes and Gold Mined by Pit....5
Table 1-4: Industrial Heap Leach Metal Recovery Estimates....6
Table 1-5: Weighted Average for Lime and Cement Addition from Column Tests....6
Table 1-6: Reprocessing Parameters from Spent Materials....6
Table 1-7: Weighted Average Reagent Consumption by Source....7
Table 1-8: Summary of Capital Costs....8
Table 1-9: Summary of Base Case Operating Cost Estimates for the Brewery Creek Operation....9
Table 1-10: Summary of Economic Modelling Results....9
Table 1-11: Post tax and royalty sensitivities for various gold prices....10
Table 2-1: Date of Qualified Person's Site Visit....13
Table 4-1: Brewery Creek Quartz Claims and Licenses....18
Table 5-1: Haul Road Lengths and Condition of Road....23
Table 5-2: Golden Predator Anticipates Rebuilding and Activating the Following Ponds for Production....24
Table 6-1: Summary of Historical Drilling....28
Table 10-1: Summary of Drilling Conducted by Golden Predator, 2009-2012....50
Table 10-2: Summary of Drilling for Resource Estimate Areas....55
Table 11-1: Historical Analytical Laboratories and Methods....57
Table 11-2: List of QA/QC control and Standard Reference Materials from 2012....62
Table 11-3: Summary of SRM's Submitted in 2012....64
Table 11-4: 2012 Duplicate Sample Statistics....66
Table 11-5: Summary of Available Standard Sample Results....67
Table 11-6: Summary of Blank Sample Results....69
Table 11-7: Summary of Duplicate Samples....71
Table 12-1: Summary of Drillhole Information Available for Review, Effective March 11, 2012....74
Table 12-2: Recent Twins of Historical Drillholes....76
Table 12-3: Independent Drill Core Samples Collected by Tetra Tech EBA....78
Table 12-4: Drill Core Samples Examined by RMI....80
Table 12-5: Drillhole Assay Samples Verified by RMI....80
Table 12-6: Independent Sample Results....88
Table 13-1: Drill Core Composites Gold Head Assay Results....89
Table 13-2: Drill Core Composites Head Screen Analyses....90
Table 13-3: Drill Core Composites Bottle Roll Tests....91
Table 13-4: Drill Core Composites Column Leach Tests Physical Characteristics....94
Table 13-5: Drill Core Composites Column Leach Tests....95
Table 13-6: Heap Leach Design Parameters....100
Table 13-7: Column Leach Tests and Modeled Gold Extraction....100
Table 13-8: Weighted Average for Lime and Cement Additions from Column Tests....101
Table 13-9: Weighted Average for Lime and Cement Additions from Column Tests....101
Table 13-10: Sodium Cyanide Consumption....102
Table 13-11: Reprocessed Ore Physical Characteristics....104
Table 13-12: Reprocessed Ore Column Leach Tests....104
Table 13-13: Reprocessing Parameters from Spent Material....105
Table 13-14: Weighted Average Extraction and Reagent Consumption....105
Table 13-15: Work Index and Abrasion Results....107

Table 14-1: Sample Gold Assay Statistics (Gold Grades Reported in g/t) 110
Table 14-2: Block Model Parameters 111
Table 14-3: Gold Cap Values 111
Table 14-4: Composite Gold Assay Statistics (Gold grades reported in g/t) by Zone 112
Table 14-5: Block Estimation Parameters 113
Table 14-6: Structural Domain Estimation Parameters 114
Table 14-7: Descriptive Statistics for Gold in Composite Samples and Model 115
Table 14-8: ID3 and NN Model Comparison 120
Table 14-9: Drillhole Data 123
Table 14-10: Drillhole Assay Statistics – Gold Fire Assays by Area 123
Table 14-11: Drillhole Assay Statistics – Gold Fire Assays by Sample Type 124
Table 14-12: Drillhole Assay Statistics – Gold Fire Assays by Major Rock Type 125
Table 14-13: Drillhole Assay Statistics – Gold Fire Assays by Logged Oxidation 125
Table 14-14: Drillhole Assay Statistics – Cyanide Soluble Gold Assays by Area 126
Table 14-15 : Drillhole Assay Statistics – Gold Preg Rob Assays by Area 127
Table 14-16: Bulk Density for BH, SC, FS, WB, and EB Models 128
Table 14-17: Block Model Extents 129
Table 14-18: Gold Grade Capping Limits by Area 130
Table 14-19: Gold Grade Estimation Constraints 135
Table 14-20: Gold Grade Estimation Parameters 136
Table 14-21: Global Bias Check – Inverse Distance vs. Nearest Neighbor Grades 142
Table 14-22: Descriptions for Major Rock Types used in Geologic Model 148
Table 14-23: Drillholes by deposit used in the Mineral Resource Estimate 149
Table 14-24: Specific Gravity Values used for North Slope and Sleeman 150
Table 14-25: Block Model Origins and Dimensions 151
Table 14-26: High Grade Caps Applied to Composites 151
Table 14-27: Summary Descriptive Statistics for Raw Assay and 2 m Uncapped Composite Samples 152
Table 14-28: Summary of Variogram Parameters 152
Table 14-29: Summary of Search Ellipse Parameters 153
Table 14-30: Resource Block Model Comparative Statistics 157
Table 14-31: Oxide Pit Parameters 160
Table 14-32: Summary of Total Mineral Resources – Oxide Mineral Resources 161
Table 14-33: Summary of Total Mineral Resources – Sulfide Mineral Resources 162
Table 16-1: Process Costs and Recoveries Used in Pit Optimization 163
Table 16-2: Results of RMR Analysis 164
Table 16-3: Joint Sets for the Identified Domains 165
Table 16-4: Summary of Kinematic Bench Face Instability Analyses 166
Table 16-5: Summary of Limit Equilibrium Analyses 167
Table 16-6: Slope Design Parameters for Fosters Pit 173
Table 16-7: Ktonnes (Kt) Processed from the Pits and Reprocessed from the existing Heap Leach Pad for Each Year of Operations 182
Table 16-8: Gold mined in kg from Pits and Old Heap Leach Over Life of Mine 183
Table 16-9: Brewery Creek Annual Waste Mining Schedule in Kt 184
Table 16-10: Input Assumptions for Physical Characterization of Rock Type 185
Table 16-11: Input Assumptions for Embankment and Footprint Dimensions 185
Table 16-12: Allocation of Waste Material from each Open Pit to each Dump Location in LCM (x10^3) 185
Table 16-13: Waste Rock Types Placed in each Conceptual Waste Rock Dump or Open Pit Backfill 186
Table 16-14: Mining Mobile Machinery Selected 197

Table 16-15: Design Speed vs Radii 199
Table 16-16: Haul Truck Cycle Times from the Pits to Process Stockpile 199
Table 16-17: Haul Truck Cycle Times for Waste Rock from Pits to Waste Rock Dump 200
Table 16-18: Mining Related Equipment Hours, Consumable Quantities and Labour Numbers Over Life of Mine 201
Table 17-1: Heap Leach Pad Design Criteria 202
Table 17-2: Heap Leach Geotechnical Parameters 206
Table 17-3: Results of Analysis of Yearly Pond Maximum Volumes over duration of Operation 208
Table 18-1: Current Buildings at Brewery Creek 228
Table 18-2: Cut and Fill Slopes for Project Infrastructure Areas 237
Table 18-3: Power Distribution for Brewery Creek Mine Project 245
Table 18-4: Grid Power Options and Estimated Costs (source BBA Engineering) 246
Table 20-1: Release Mechanisms by Rock Type 249
Table 20-2: Brewery Creek Hydrometric Stations 250
Table 20-3: Predicted Mean Monthly Runoff During an Average Precipitation Year 251
Table 21-1: Summary of Capital Costs 254
Table 21-2: Preproduction Pit Development and Haul Road Cost Estimate 255
Table 21-3: Capital Cost Estimate for Mining Equipment 255
Table 21-4: Estimated Capital for Mine Site Infrastructure 256
Table 21-5: Earthworks Rates Used for the Capital Cost Estimate 257
Table 21-6: Direct Labour Costs 258
Table 21-7: Unit Rates and Labour Multipliers for the Structural Steel Construction Costs 259
Table 21-8: Initial Capital 260
Table 21-9: Sustaining Capital 261
Table 21-10: Engineering Drawings 263
Table 21-11: Mobile Equipment 265
Table 21-12: Heap Leach Cells 8 to 10 Construction Costs 265
Table 21-13: Summary of Base Case Operating Cost Estimates for the Brewery Creek Operation 268
Table 21-14: Breakdown of G and A Cost Estimate for Brewery Creek over LOM 268
Table 21-15: Salaries Used in G and A Calculation 269
Table 21-16: Breakdown of Mining Costs By Allocation and By Activity 270
Table 21-17: Estimated Reagent and Fuel Costs 272
Table 21-18: Reagents 273
Table 21-19: Fuel 273
Table 21-20: Power 273
Table 21-21: Labour 274
Table 21-22: Other Consumables 274
Table 21-23: Processing Plant Annual Operating Costs 274
Table 22-1: Summary of Economic Modelling Results 275
Table 22-2: Summary of Undiscounted Production and Economic Results 276
Table 22-3: Summary of Taxes and Royalties included in the Financial Model 277
Table 22-4: Post Tax and Royalty Sensitivities for Various Gold Prices 278
Table 22-5: Results of the Trade-Off Studies 279
Table 22-6: Results of the Technical Economic Analysis 280
Table 24-1: Milestones Time Line 283

LIST OF FIGURES IN TEXT

Figure 4–1: Locality Map for Brewery Creek, Dawson City, Yukon Territory 15
Figure 4–2: Quartz Claims and Quartz Mining Leases 16
Figure 4–3: Quartz Claims and Property Royalty Agreements 17
Figure 5–1: Overview of the current reclaimed heap leach facility and solution ponds (Golden Predator, 2012) 25
Figure 6–1: Compilation of Historical Gold in Soil Results (from Diment and Simpson, 2009) 27
Figure 6–2: Trench and Drillhole Locations (Diment and Simpson, 2009) 30
Figure 7–1: Regional Geology Map 32
Figure 7–2: Regional Geology Legend 33
Figure 7–3: Property Geology Map 35
Figure 7–4: Property Geology Legend 36
Figure 7–5: Photo from helicopter of the Brewery Creek Property, looking east with reclaimed Canadian and Kokanee deposits along left side and Laura Creek along valley bottom (Golden Predator, 2012) 45
Figure 8–1: Geological Model Schematic (Extracted and Modified from Lindsay, 2006) 47
Figure 9–1: Airborne Magnetics, Total Magnetic Intensity over Brewery (Precision, 2012) 49
Figure 10–1: Property Drilling Map 51
Figure 11–1: Core Logging Facility, March 2012 59
Figure 11–2: Core and Sample Processing Facility, March 2012 60
Figure 11–3: Onsite Core Cutting Equipment, March 2012 60
Figure 11–4: Performance of 2012 Blanks 63
Figure 11–5: Performance of 2012 SRM CDN-GS-1P5D 64
Figure 11–6: Performance of 2012 SRM CDN-GS-P3B 65
Figure 11–7: Performance of 2012 SRM CDN-GS-P4A 65
Figure 11–8: Performance of 2012 SRM CDN-GS-5G 66
Figure 11–9: 2012 Duplicate Sample QQ Plot 67
Figure 11–10: Blank Gold Assay Data, if Detected 5-Times above RL 69
Figure 11–11: Blank Gold Assay Data, if Detected 5-Times above RL 70
Figure 11–12: Duplicate Sample Results 71
Figure 12–1: Decile-decile Plot Comparing 2 Metre Composites of all Core and RC Gold Data Included in this Report 75
Figure 12–2: Decile-decile Plot Comparing 2 Metre Composites of Historical Viceroy/Loki Core and RC Gold Data Included in this Report 76
Figure 12–3: Decile-decile Plot Comparing 2 Metre Composites of Recent Golden Predator Core and Historic Core Gold Data Included in this Report 77
Figure 12–4: QQ Plot Comparing Diamond Core and RC Samples - Bohemian Deposit 82
Figure 12–5: QQ Plot Comparing Diamond Core and RC Samples - Classic Deposit 82
Figure 12–6: QQ Plot Comparing Golden Predator Diamond Core and Viceroy RC Samples - Bohemian Deposit 83
Figure 12–7: QQ Plot Comparing Golden Predator Diamond Core and Loki RC Samples - Fosters Deposit 84
Figure 12–8: QQ Plot Comparing Golden Predator Diamond Core and Viceroy RC Samples - East Big Rock Deposit 84
Figure 12–9: QQ Plot Comparing Golden Predator RC and Viceroy RC Samples - East Big Rock Deposit 85
Figure 12–10: Core vs. RC Samples 86
Figure 13–1: Bottle Roll Head Assays 93
Figure 13–2: Bottle Roll Gold Extraction Results 93

Figure 13–3: West Big Rock Column Leach Gold Extraction 97
Figure 13–4: East Big Rock Column Leach Gold Extraction 98
Figure 13–5: Lower Fosters Column Leach Gold Extraction 98
Figure 13–6: Bohemian Column Leach Gold Extraction 99
Figure 13–7: Schooner Column Leach Gold Extraction 99
Figure 14–1: Locations of the fifteen block models as purple polygons 108
Figure 14–2: Lucky Area Contact Plot 111
Figure 14–3: KOGD Area Raw Au Sample Cumulative Probability Plot 112
Figure 14–4: Pairwise Relative Experimental Variograms within Horizontal and Vertical Directions for all Samples within SDMN1 113
Figure 14–5: Pairwise Relative Experimental Variograms within Horizontal and Vertical Directions for all Samples within SDMN2 113
Figure 14–6: Validation Section A-A' of Kokanee Block Model (See Figure 14–8 for Location) 116
Figure 14–7: Validation Section B-B' of Golden Block Model (See Figure 14–8 for Location) 116
Figure 14–8: Detailed Drillhole Locations of KOGD and Location of Cross Sections A-A' and B-B' 117
Figure 14–9: Validation Section C-C' of Pacific Block Model (See Figure 14–11 for Location) 117
Figure 14–10: Validation Section D-D' of BL Block Model (See Figure 14–11 for Location) 118
Figure 14–11: Detailed Drillhole Locations of PABL and Location of Cross Sections C-C' and D-D' 118
Figure 14–12: Validation Section E-E' of Lucky Block Model (See Figure 14–13 for Location) 119
Figure 14–13: Detailed Drillhole Locations of LU and Location of Cross Section E-E 119
Figure 14–14: Au Solubility vs. Logged Oxidation (BH Intrusives) 122
Figure 14–15: Au Solubility vs. Logged Oxidation (SC Intrusives) 122
Figure 14–16: Au Probability Plot – (Bohemian Intrusives) 130
Figure 14–17: Au Grade Correlogram – (Oxidized Bohemian Intrusives) 131
Figure 14–18: Au Grade Correlogram – (Oxidized Schooner Intrusives) 132
Figure 14–19: Au Grade Correlogram – (Oxidized Lower Fosters Intrusives) 132
Figure 14–20: Au Grade Correlogram – (Oxidized West Big Rock Intrusives) 133
Figure 14–21: 0.2 g/t Au Indicator Correlogram – (Oxidized Bohemian Intrusives) 133
Figure 14–22: 0.2 g/t Au Indicator Correlogram – (Oxidized Schooner Intrusives) 134
Figure 14–23: 0.2 g/t Au Indicator Correlogram – (Oxidized Lower Fosters Intrusives) 134
Figure 14–24: 0.2 g/t Au Indicator Correlogram – (Oxidized West Big Rock Intrusives) 135
Figure 14–25: Bohemian Block Model Section A-A' (See Figure 14 27 for Location) 137
Figure 14–26: Schooner Block Model Section B-B' (See Figure 14 27 for Location) 137
Figure 14–27: Location of Bohemian (A-A') and Schooner (B-B') Cross Sections 138
Figure 14–28: Lower Fosters Block Model Section C-C' (See Figure 14 29 for Location) 138
Figure 14–29: Location of Lower Fosters (C-C') Cross Section 139
Figure 14–30: West Big Rock Block Model Section D-D' (See Figure 14 32 for Location) 139
Figure 14–31: East Big Rock Block Model Section E-E' (See Figure 14 32 for Location) 140
Figure 14–32: Location of West Big Rock (D-D') and East Big Rock (E-E') Cross Sections 140
Figure 14–33: Classic Block Model Section F-F' (See Figure 14 34 for Location) 141
Figure 14–34: Location of Classic (F-F') Cross Section 141
Figure 14–35: Bohemian Gold Swath Plot by Elevation Levels 142
Figure 14–36: Schooner Gold Swath Plot by Elevation Levels 143
Figure 14–37: Lower Fosters Gold Swath Plot by Elevation Levels 143
Figure 14–38: West Big Rock Gold Swath Plot by Elevation Levels 144
Figure 14–39: East Big Rock Gold Swath Plot by Elevation Levels 144
Figure 14–40: Classic Gold Swath Plot by Elevation Levels 145
Figure 14–41: Lone Star Gold Swath Plot by Elevation Levels 145

Figure 14–42: Plan View of Viceroy Heap Leach Pad Showing Sonic Holes 147
Figure 14–43: Heap Leach Cross Section A-A' 147
Figure 14–44: Specific Gravity Determination by Rock Type 150
Figure 14–45: Oblique Section of North Slope Gold Grade Model (40 metre wide) 153
Figure 14–46: Oblique Section of Sleeman Gold Grade Model (25 metre wide) 154
Figure 14–47: Grade Tonnage Curve for the North Slope Deposit 155
Figure 14–48: Grade Tonnage Curve for the Sleeman Deposit 155
Figure 14–49: Legend for grade-tonnage curve line types 156
Figure 14–50: Cumulative Probability Plot for North Slope Deposit 158
Figure 14–51: Cumulative Probability Plot for Sleeman Deposit 158
Figure 16–1: Design of Single Lane Haul Roads 171
Figure 16–2: Design of Dual Lane Haul Roads 172
Figure 16–3: West Big Rock Ultimate Pit Design 174
Figure 16–4: East Big Rock Ultimate Pit Design 175
Figure 16–5: Fosters Ultimate Pit Design 176
Figure 16–6: Kokanee Ultimate Pit Design 177
Figure 16–7: Golden Ultimate Pit Design 178
Figure 16–8: Lucky Ultimate Pit Design 179
Figure 16–9: Bohemian Ultimate Pit Design 180
Figure 16–10: Schooners Ultimate Pit Design 181
Figure 16–11: Waste Rock Storage Facility CWD – 01 187
Figure 16–12: Waste rock storage facility CWD – 03 188
Figure 16–13: Waste rock storage facility CWD – 04 189
Figure 16–14: Waste rock storage facility CWD – 06 190
Figure 16–15: Waste rock storage facility CWD – 07 191
Figure 16–16: West Big Rock Backfill 192
Figure 16–17: Lower Fosters Backfill 193
Figure 16–18: Kokanee Backfill 194
Figure 16–19: Golden Backfill 195
Figure 16–20: Schooner Backfill 196
Figure 17–1: Heap Leach Liner Layering 208
Figure 17–2: Conceptual Process Plant Flowsheet 212
Figure 17–3: Conceptual Crushing Plant (SGS 67-F-001, 2014) 214
Figure 17–4: Conceptual Agglomeration Flowsheet (SGS 15-F-001, 2014) 216
Figure 17–5: Conceptual Solution Management Flowsheet (SGS 45-F-001, 2014) 218
Figure 17–6: Conceptual Carbon Absorption Flowsheet (SGS 50-F-001, 2014) 220
Figure 17–7: Conceptual Stripping and Refining Flowsheet (SGS 55-F-001, 2014) 222
Figure 17–8: Conceptual Acid Washing and Carbon Reactivation Flowsheet (SGS 55-F-002, 2014) 224
Figure 17–9: Conceptual Heap Detoxification Flowsheet (SGS 61-F-001, 2014) 226
Figure 18–1: Brewery Creek Overall Site Layout 230
Figure 18–2: Brewery Creek Site Plan Showing Haul roads and pit locations 231
Figure 18–3: Brewery Creek Crushing facility 232
Figure 18–4: Brewery Creek Processing facilities 233
Figure 18–5: Brewery Creek Camp Area 234
Figure 22–1: Sensitivity Chart for Post-Tax NPV 278
Figure 24–1: Projected Brewery Creek Development Schedule 284

ACRONYMS & ABBREVIATIONS

AAS	atomic absorption spectroscopy
AQM	Altered Quartz Monzonite
ARG	Graphitic Argillite
AU	Gold
BC	British Columbia
BCRT	Brewery Creek Reserve Trend
BFA	bench face angles
BRC	Brewery Creek
C	Carbon
CNCF	cumulative net cash flow
DCF	discounted cash flow
DGPS	differential global positioning system
GCL	Geosynthetic Clay Liner
Golden Predator	Golden Predator Canada Corp
HQ Drilling	drill diameter of 63.5 mm
ICP-AES	Inductively Coupled Plasma - Atomic Emission Spectroscopy
IRR	internal rate of return
LAQM	Limonitic Altered Quartz Monzonite
METSIM	metallurgical software program
MLI	McClelland Laboratories Inc.
MMTS	Moose Mountain Technical Services
NCF	net cash flow
NSR	net smelter royalty
PFS	Prefeasibility Study
PGA	peak ground acceleration
PQ Drilling	drill diameter of 85 mm
Preg	Pregnant solution
PVC	polyvinyl chloride
QA/QC	quality assurance and quality control
QP	qualified person
RC Drilling	reverse circulation drilling
RMI	Resource Modeling Inc.
RMR	rock mass rating
ROM	run of mine
RQD	rock quality designation
SED_NX	unoxidized sediments
SED_OX	oxidized sediments
SGS	SGS Metcon
Std.	standard
Tetra Tech EBA	Tetra Tech EBA Inc.

UNITS AND SYMBOLS

$	US dollar
$/t	dollars per tonnes/m³
%	percent
°	degrees
µm	micron
A	Amperes
CAD$	Canadian dollar
CAD$/tonne	Canadian dollars per tonne
cm	centimetre
ft.	feet
ft.²	square feet
g	metric gram
g/cc	grams per cubic centimetre
g/t	metric gram per metric tonne
gpt	metric gram per metric tonne
gpm/ft²	gallons per minute per square foot
ha.	hectare
HP	horse power
hr.	hour
kg	metric kilogram
Kg/t	kilograms per tonne
Kl/t	kilo litres per tonne
km	kilometre
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt
KWhr	kilowatt hour
m	metres
m²	metres squared
m³	metres cubed
mins	minutes
ml	millilitre
mm	millimetre
MVA	megavolt-ampere
MW	megawatt
NaCN	sodium cyanide
Oz.	ounce
ppm	parts per million
sf.	square feet
Ton	imperial ton
tonne	metric tonne
tpd	tonnes per day
V	volt
yd.	yards
yd.³	cubic yards

1.0 EXECUTIVE SUMMARY

1.1 Introduction

Golden Predator Exploration Ltd. (Golden Predator) is a Canadian exploration company listed on the TSX Venture Exchange as GPY, and main office located in Edmonton, Alberta. The company is evaluating the economics of resuming mining operations at their wholly owned Brewery Creek Property, Yukon, through a combination of open pit mining and reprocessing spent heap leach material for gold doré recovery.

In August 2012, Golden Predator Canada Corp. (Golden Predator) commissioned a team of engineering consultants to complete a Preliminary Economic Assessment (PEA) to be prepared in accordance with NI 43-101 disclosure standards. The contributors to the PEA include: Tetra Tech EBA Inc. (Tetra Tech EBA) of Vancouver, BC; Tetra Tech Inc. of Tucson, AZ; SGS - E&S Engineering Solutions Inc. (SGS) of Tucson, AZ; Resource Modeling Inc. (RMI) of Stites, Idaho; Gustavson and Associates of Lakewood (Gustavson), Colorado; and Access Consulting Group (Access) of Whitehorse, YT. The PEA is based on exploration drilling results, certified laboratory analysis of samples and subsequent analysis of the results. The Effective Date of the study is July 22nd, 2014.

1.2 Property Description and Location

The Brewery Creek Property (the Property) includes a past producing heap leach gold mining operation, located approximately 55 km due east of Dawson City in the northwestern Yukon. The property is centered at Latitude 64.041887° N and Longitude 138.206389° W or UTM NAD83 Zone 7N at 636401(m) E, 7104673(m) N. The Property consists of total 1,075 quartz claims (93 of which have been converted to mining leases) covering approximately 181 km^2.

The site is accessible year round from the Klondike Highway connecting Whitehorse and Dawson City. Access to the site requires travelling 8 km along the Dempster Highway, which intersects the Klondike Highway about 40 km from Dawson City. About 20 km of well-maintained gravel road connects the mine site to the Dempster Highway.

The property was previously mined and operated by Viceroy Minerals Corporation (Viceroy) between 1995 and 2002, with approximately 280,000 ounces of gold produced.

Although much of the infrastructure from past mining operations has been removed or rehabilitated, the original administration building, with an office, core logging facility, warehouse and storage still exists on site along with the heap leach pad, process and overflow ponds. This infrastructure has been on site since 1996 when Viceroy commenced operations at the mine. In 2011, Golden Predator added several mobile accommodation trailers and wall tents to accommodate an increase in staff and personnel. A network of roads exists that connect various exploration sites and work locations to the main camp. Previously reclaimed roads have been brought back into use and are maintained by Golden Predator.

Golden Predator owns a 100% interest in the Brewery Creek Property, subject to a 2% net smelter return royalty (“NSR”) in favour of Alexco Resource Corp. (Alexco) on the first 600,000 ounces of gold produced from the Property, after which the NSR will increase to 2.75%. Golden Predator has the right to repurchase 0.625% of the increased NSR for CAD $2,000,000 (which, if so acquired, would result in a 2.125% NSR on gold to Alexco).

In exchange for a 100% interest in the Brewery Creek Property and in addition to the NSR, the Company paid CAD $3,205,000 to Alexco, representing the cash consideration to be paid under the Purchase Agreement (CAD $4,000,000) less the amount of the reclamation bond that had been posted by Alexco with the Yukon government (CAD $795,000). The current quartz mining licence and water licence have been transferred to Golden Predator from Alexco.

1.3 Property History

Historical exploration surveys conducted at Brewery Creek between 1988 and 2006 included geological mapping, extensive grid soil sampling, ground and airborne geophysical studies, mechanized surface trenching, and extensive core and reverse-circulation drilling.

Viceroy ran an operating mine at Brewery Creek between 1996 and 2002. During this period approximately 280,000 ounces of gold were produced from seven near-surface oxide deposits occurring along strike within the historically termed “Brewery Creek Reserve Trend” (BCRT). The first gold pour at the Brewery Creek Mine was completed on November 15, 1996 with 10,175 ounces being produced prior to commencement of full commercial production in May 1997. During 1997, a total of 72,387 ounces of gold were produced at a cash cost of $USD 184 per ounce of gold. In 1998 production totaled 79,396 ounces at a cash cost of $USD 177 per ounce. Production in 1999 fell to 48,164 ounces while operating costs rose to a cash cost of $USD 288 per ounce of gold. Viceroy suspended seasonal mining operations earlier than planned and hired an independent consulting company to study processes in an effort to improve recoveries. In 2000, Viceroy concentrated on selectively mining the mineralization containing the highest grades. Production in 2000 fell to 48,048 ounces of gold at a cash operating cost of $USD 243 per ounce of gold. Mining ceased in 2001, but heap leaching continued with production of 18,542 ounces of gold at a cash operating cost of $USD 222 per ounce of gold.

During 2002, Viceroy undertook and completed reclamation consisting of re-contouring and re-vegetation of pits and dumps. A final closure and decommissioning plan was prepared and submitted as required, to the Yukon regulatory agencies, with the primary elements of the plan adopted as water license amendments granted in April 2005.

1.4 Geological Setting and Mineralization

The Brewery Creek Property is located within the foothills of the Ogilvie Mountains along the northeastern boundary of the Tintina Trench. The Tintina Trench forms a 15-kilometre wide erosional valley formed by the northwest-striking Mesozoic to Tertiary Tintina Fault. In the vicinity of the Property, the Tintina Fault juxtaposes Selwyn Basin stratigraphy to the northeast against accreted terranes of the Canadian Cordillera to the southwest. Selwyn Basin stratigraphy is composed of Late Proterozoic and Paleozoic marginal basin deposits of ancient North America. The Cordillera rocks are dominantly composed of Klondike Schist and other allied rocks of the Yukon-Tanana Terrain, an allochthonous terrain of primarily volcanic arc rocks that evolved in mid to late Paleozoic time.

The Brewery Creek Project is in Selwyn Basin rocks northeast of the Tintina Trench. The local stratigraphy consists of late Proterozoic to Paleozoic marginal basinal and platformal clastic and pelitic lower greenschist facies metasedimentary rocks. The provenance of the protoliths was the North American Craton. The stratigraphy includes thick sequences of Lower Proterozoic Hyland Group, Cambrian-Ordovician Road River Group and Devonian-Mississippian Earn Group sedimentary rocks.

The Selwyn Basin rocks have been polydeformed and imbricated by the Jura-Cretaceous Dawson, Tombstone and Robert Service Thrusts. The Hyland, Road River and Earn Group rocks are cut by Cretaceous intrusive units (Tombstone Plutonic Suite) that form a northwest-trending belt of widely spaced intermediate to siliceous stocks and plutons that closely parallel the Tintina Trench. In the Brewery Creek area, these igneous rocks are comprised of monzonite and quartz monzonite that primarily intruded along the thrust faults and formed sill-like geometries.

Gold mineralization at Brewery Creek is predominantly hosted within or adjacent to the felsic intrusive rocks. Gold is associated with carbonate/clay, quartz and pyrite/arsenopyrite alteration of monzonite/quartz monzonite intrusive rocks and adjacent siliciclastic rocks.

1.5 Exploration

Exploration conducted by Golden Predator includes geophysical surveys, soil sampling surveys and an extensive drilling campaign. These surveys were undertaken to define the limits of known mineralized zones and examine previously untested parts of the Property. In 2011, Precision GeoSurveys Inc. of Vancouver, BC was contracted to fly an airborne magnetic survey. The survey was done in order to better define the magnetic signatures in known areas of mineralization and to investigate these same signatures in unexplored areas. During this time, Golden Predator also completed soil sampling of the Classic zone and on new extensional claims on the property, as well as an IP survey over the Sleeman zone in the eastern part of the property.

1.6 Mineral Resources

Mineral resource estimates have previously been reported by Golden Predator for a total of fifteen individual deposits (including the spent ore on the historic leach pad) on the Brewery Creek Property. The most recent estimates were disclosed in a press release by Americas Bullion Royalty Corp. on September 19, 2013 and a supporting by technical report titled "NI 43-101 Technical Report on Resources, Brewery Creek Project, Yukon, Canada" filed on Sedar on October 23, 2013. These estimates are unchanged and remain current for use in the PEA. The current oxide mineral resources for the Brewery Creek Property are summarized in Table 1-1, and sulfide resources in Table 1-2.

Table 1-1: Summary of Oxide Mineral Resources

Resource Area	Au Cut-off (g/t)	Indicated Oxide Resources			Inferred Oxide Resources		
		Tonnes (000)	Au (g/t)	Au Ozs (000)	Tonnes (000)	Au (g/t)	Au Ozs (000)
Kokanee	0.54	1,201	1.19	46	279	1.19	11
Golden	0.54	1,070	1.38	47	247	1.25	10
Pacific	0.53	373	1.01	12	131	0.91	4
Blue	0.53	250	1.29	10	29	0.98	1
Lucky	0.54	2,394	1.36	105	236	1.27	10
Bohemian	0.49	1,491	1.31	63	134	1.49	6
Schooner	0.51	1,108	1.99	71	243	2.65	21
Lower Fosters	0.51	1,090	1.61	56	492	1.52	24
West Big Rock	0.45	722	1.27	29	38	0.75	1
East Big Rock	0.48	596	1.10	21	21	0.87	1
Classic	0.54	-	-	-	3,711	0.81	97
Lone Star	0.54	-	-	-	1,522	0.88	43
North Slope	0.5	756	1.15	28	412	1.05	14
Sleeman	0.5	124	1.14	5	132	0.84	4
Historical Viceroy Pad	0.30	2,977	0.88	84	1,682	0.60	32
Total		**14,152**	**1.27**	**577**	**9,309**	**0.93**	**279**

Note: Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an inferred resource will ever be upgraded to a higher category. Tonnes and contained gold have been rounded to the nearest thousand.

* cut-off grades are based on a gold price of $1,250 per ounce (below 3 year trailing average) and recovery rate of 70 to 83% depending on the deposit.

Table 1-2: Summary of Sulfide Mineral Resources

Resource Area	Au Cut-off (g/t)	Indicated Sulfide Resources			Inferred Sulfide Resources		
		Tonnes (000)	Au (g/t)	Au Ozs (000)	Tonnes (000)	Au (g/t)	Au Ozs (000)
Kokanee	0.70	-	-	-	1,547	1.33	66
Golden	0.70	-	-	-	649	1.20	25
Pacific	0.70	-	-	-	707	1.45	33
Blue	0.70	-	-	-	1,358	1.31	57
Lucky	0.70	-	-	-	1,783	1.36	78
Bohemian	0.70	-	-	-	973	1.58	50
Schooner	0.70	-	-	-	313	1.42	14
Lower Fosters	0.70	-	-	-	883	1.45	41
West Big Rock	0.70	-	-	-	381	1.28	16
East Big Rock	0.70	-	-	-	170	1.00	5
Classic	0.70	-	-	-	-	-	-
Lone Star	0.70	-	-	-	-	-	-
North Slope	0.70	2,122	1.26	86	2,686	1.36	118
Sleeman	0.70	1,337	1.30	56	958	1.40	43
Total		**3,459**	**1.28**	**142**	**12,408**	**1.37**	**546**

Note: Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an inferred resource will ever be upgraded to a higher category. Tonnes and contained gold have been rounded to the nearest thousand.

* cut-off grades are based on a gold price of $1,250 per ounce (below 3 year trailing average) and recovery rate of 70 to 83% depending on deposit.

1.7 Mining

For the purpose of the PEA, mining is planned to be carried out using open pit truck and shovel methods. Eight deposits were evaluated further for mining namely, from West to East, West Big Rock, East Big Rock, Lower Fosters, Kokanee, Golden, Lucky, Bohemian and Schooner.

Mining and recovery of material from the old heap leach has also been considered, based on the fact that the material placed on the old heap leach was not crushed and that recent metallurgical work has shown that crushing and reprocessing this material will result in recoveries of an estimated 45%. This material has been scheduled based on making up tonnage from the pits, when the process feed from the pits is not planned to be mined at full capacity.

To evaluate the possibility of undertaking open pit mining, the resource block models were imported into a pit optimisation software package called Geovia Whittle 4.5™. Criteria were applied to the mining and processing of mineralised blocks and waste blocks that would be mined. Pit selection was based on maximising the value obtained from mining the pits. The resulting pit shells were imported into a general mining package to create open pit designs with benched slopes and pit access ramps.

The design pits were then scheduled on the basis of the contained rock types and mineralisation within the design pits. Additional process feed was also considered for mining from the old heap leach pad in accordance with the resource evaluated for the material on the pad.

Table 1-3 shows a summary of the pits proposed to be mined over the life of mine.

Table 1-3: Total Tonnes and Gold Mined by Pit

Pit Area	Process Feed Mined Over Life of Mine (kt)	Waste Rock Mined (kt)	Gold Mined (kg)	Gold grade (gpt)
Schooner	1,044	8,198	2,157	2.07
Fosters	1,275	5,599	2,067	1.62
Bohemian	1,577	4,960	1,919	1.22
Golden	878	2,776	1,176	1.34
Kokanee	1,243	3,908	1,321	1.06
WBR	809	4,167	945	1.17
EBR	465	2,235	496	1.07
Lucky	2,973	11,677	3,764	1.27
Total from Pits	**10,264**	**43,520**	**13,845**	**1.35**
Total from Old Heap Leach	**4,180**	**3,366**	**3,219**	**0.77**

Waste will be stored in waste dumps, which have been conceptually designed for the PEA at locations, which have been found suitable, based on geotechnical investigations. Where possible, waste will be placed in mined out pits to reduce closure costs and reduce the project footprint.

Mining equipment is estimated to cost US$17.8 million to purchase. For the PEA, leasing of mining equipment has been considered, this reducing capital costs but increasing operating costs.

The mining equipment includes equipment for drilling blast holes, loading of haul trucks (by loader and shovel) and a total of 10 trucks for hauling. Support equipment includes bulldozers, pit service loaders, grader, ANFO truck, medium trucks, tire handlers and light vehicles. Equipment for maintaining haul roads has also been included.

1.8 Electrical Power

The power required for the Brewery Creek Property will be supplied by on-site generation using diesel fuel. Power supply is also available from the utility through Yukon Energy Corporation (YEC), via a 27 km long, 69 kV utility transmission line from Dempster Corner though this has not been considered for the PEA. The Brewery Creek Mine Substation will include an electrical room, housing the main 4.16 kV switchgear for distributing power on site, and a power generation building to house the generators. .

1.9 Mineral Processing and Metallurgical Test Work

Considerable metallurgical testing of material from the Brewery Creek Property were conducted and presented in earlier reports completed in 1988 by Loki Gold Corporation. The test work was conducted by Kappes, Cassiday & Associates and Lakefield in the 1990s. Current metallurgical testing of both new resource and historical leach pad material has been done by McClelland. A variability study was also conducted on new materials, which included, bottle roll tests, column tests, column lock-cycle tests, screen and head analysis.

Metallurgical data have been extracted from a McClelland draft report, MLI Job No. 3618, to estimate metallurgical performance of residue material from the existing heap leach pad. MLI Job No. 3719 was used to estimate performance of new mined materials from potential pit sites. Gold extraction was modelled for five target pits and reagent consumptions were calculated from test work and weighted based on the minable pit tonnage.

Industrial scale-up was applied to each target pit and a calculation of the metal produced conducted.

Table 1-4: Industrial Heap Leach Metal Recovery Estimates

	Gold Extraction by Deposit (%)				
	West Big Rock	**East Big Rock**	**Lower Foster**	**Bohemian**	**Schooner**
Cumulative Extraction					
30 days (0.7 kl/t)	81.3	75.4	72.8	78.4	73.6
60 days (1.3 kl/t)	86.6	80.6	76.7	81.0	77.6
90 days (2.0 kl/t)	87.3	81.2	77.6	81.2	78.1
Discount for Industrial Practice	3.5	3.5	3.5	3.5	3.5
Heap Leach Average Extraction	83.8	77.7	73.2	77.7	74.6
CIC/Goldroom Recovery	99.0	99.0	99.0	99.0	99.0
Gold Recovery to Doré	82.9	77.0	72.5	77.0	73.9

Reagent consumption is derived from the metallurgical test work.

Table 1-5: Weighted Average for Lime and Cement Addition from Column Tests

Ore Zone	Reagent Consumption (kg/t)	
	Lime	**Cement**
West Big Rock	3.87	-
East Big Rock	3.30	-
Lower Fosters	1.73	2.00
Bohemian	3.00	-
Schooner	2.53	-

Parameters for reprocessing the historical leach pad are derived from McClelland Report on Job No. 3618, mining requirements and operational data from the original operation.

Table 1-6: Reprocessing Parameters from Spent Materials

Description	Value	Units	Comments
Resource	4,208,000	t	Estimated
Grade	0.68	g/t	Mine Plan grade
Recovery	45	%	Test work
Lime Consumption	0	kg/t	Column Leach Tests
Cyanide Consumption	0.265	kg/t	Operational data
Cement Consumption	5.75	kg/t	Column Leach Tests

The combined parameters are summarized in Table 1-7.

Table 1-7: Weighted Average Reagent Consumption by Source

Ore Zone	Au Extraction (%)	NaCN (kg/t)	Lime (kg/t)	Cement (kg/t)
West Big Rock	82.9	0.30	3.87	0
East Big Rock	77.0	0.73	3.30	0
Lower Fosters	72.5	0.23	1.73	2.00
Bohemian	77.0	0.31	3.00	0
Schooner	73.9	0.26	2.53	0
Old Heap	45.0	0.27	0	5.75
Weighted Average	**68.1**	**0.31**	**1.76**	**2.44**

1.10 Process Plant

Based on the data provided by Brewery Creek, the following process plant flow sheet has been selected:

- Crushing Plant – Tertiary crushing, modular or contracted, with primary jaw, secondary and tertiary cones, and surge bin to feed agglomeration;
- Agglomeration – Lime and cement are added to the main conveyor belt feeding the agglomeration drum. Agglomerates discharge the drum to form a crushed ore stockpile;
- Ore Stacking – Truck Stacking;
- Heap Leach Solution Management – Pumping and piping systems to circulate and collect leach liquors;
- Carbon Columns – For precious metal adsorption;
- Carbon Stripping and Refining – Concentration of gold solutions for electrowinning and production of final product;
- Acid Washing and Carbon Reactivation – Carbon handling system designed to remove acid soluble deposits on the carbon surface and a reactivation kiln to reactivate loading sites to maintain maximum gold loadings; and
- Final Detoxification – Solution detoxification for discharge.

1.11 Capital Cost Estimates

An overall capital cost of US$89 million has been estimated for the project. This includes equipment purchases, material offtakes, construction costs and labour, pre-stripping as capitalised mining costs, freight, contingencies at an overall 14 %, indirect costs and owner's costs during construction. Table 1-8 summarizes the capital costs by area.

Table 1-8: Summary of Capital Costs

Capital costs in US$000						
Capital cost item	**Estimated initial capital**	**Contingency $**	**Contingency %**	**Total Initial**	**Sustaining including contingency**	**Total capital**
Direct						
General site	$64	$3	5%	$67		$67
Site infrastructure	$2,857	$429	15%	$3,286		$3,286
Preproduction and haul roads	$890	$	0%	$890		$890
Mining equipment	$65	$16	25%	$81		$81
Mining infrastructure	$615	$92	15%	$708		$708
Total mining and site infrastructure	**$4,491**	**$540**	**12%**	**$5,031**	**$**	**$5,031**
Processing excluding heap leach construction						
Crushing	$10,902	$2,180	20%	$13,082		$13,082
Agglomeration	$3,349	$670	20%	$4,019		$4,019
Ore stacking	$728	$146	20%	$874		$874
ADR facility and heap leach equipment	$9,918	$1,984	20%	$11,901	$4,128	$16,029
Process infrastructure	$8,159	$1,632	20%	$9,790		$9,790
Total processing	**$33,055**	**$6,611**	**20%**	**$39,666**	**$4,128**	**$43,795**
Heap leach and water management						
Heap leach including ponds	$12,764	$1,915	15%	$14,679		$14,679
Water management	$83	$21	25%	$103		$103
Total HLF and water management	$12,847	$1,935	15%	$14,782	$	$14,782
Total direct	**$50,393**	**$9,087**	**18%**	**$59,480**	**$4,128**	**$63,608**
Indirect						
Capitalised mining	$11,365			$11,365		$11,365
Process indirects	$5,550	$1,110	20%	$6,661	$33	$6,694
Mining and other indirects	$1,517	$228	15%	$1,745		$1,745
Owners costs (G & A year -2 and -1)	$5,998	$	0%	$5,998		$5,998
Total indirect	**$24,431**	**$1,338**	**5%**	**$25,769**	**$33**	**$25,802**
Total capital in US$000	**$74,824**	**$10,425**	**14%**	**$85,249**	**$4,161**	**$89,410**

1.12 Operating Costs

Operating costs average US$19.95 / tonne processed over the life of mine, translating to US$ 778 per troy ounce sold. The estimation of operating costs is based on consumables, labour, maintenance and other requirements. Table 1-9 summarizes the average life of mine operating costs by area.

Table 1-9: Summary of Base Case Operating Cost Estimates for the Brewery Creek Operation

Costs			
Item	**Cost in USD$**	**Units**	**Source**
LOM average cost of mining per tonne process feed, including equipment leasing	$13.38	$/tonne	Modelled using Runge Xeras™
LOM average cost of mining process feed from old heap leach	$1.17	$/tonne	Modelled using Runge Xeras™
LOM average unit cost of mining process feed in the pits	$3.52	$/tonne	Modelled using Runge Xeras™
LOM average unit cost of mining waste rock in pits	$2.61	$/tonne	Modelled using Runge Xeras™
LOM average processing costs and placement on heap leach pad	$8.41	$/tonne	SGS
General and administrative costs per tonne process feed	$3.11	$/tonne	Estimated for each year of operation

1.13 Economics

Tetra Tech EBA prepared an economic evaluation of the Brewery Creek Project using discount cash flow modelling. The project with 9 years of operating life as proposed in the PEA has positive economics. Key economic modelling results are shown in Table 1-10.

Table 1-10: Summary of Economic Modelling Results

Summary of financial results in US$000	
Using a Gold price of US$ 1250 / oz.	
Pre-tax and royalty NPV at 5%	$45,658
Pre-tax and Royalty IRR	22%
Post Tax and Royalty NPV	**$23,315**
Post Tax and Royalty IRR	**15%**
Payback period	3.2
Using a Gold price of US$ 1300 / oz.	
Pre-tax and royalty NPV at 5%	$59,431
Pre-tax and Royalty IRR	27%
Post Tax and Royalty NPV	**$32,315**
Post Tax and Royalty IRR	**19%**
Payback period	2.9

The economic analysis is preliminary in nature and is based on the extraction of both indicated and inferred resources. Inferred resources are considered too speculative geologically to have economic considerations applied to them in order to establish mineral reserves. There is no certainty that this PEA will be realized.

The financial modelling includes consideration of all private and government royalties and taxes applicable to the property.

Tetra Tech has conducted sensitivity analysis on the PEA economic results, finding that the economics are most sensitive to gold price, followed by operating cost and then capital cost.

Table 1-11 shows the post-tax base case sensitivities at various gold prices. The break even gold price is roughly $1,135 per troy ounce.

Table 1-11: Post tax and royalty sensitivities for various gold prices

Gold Price in US$	NPV in US$000	IRR
$1,100	-$7,610	1%
$1,150	$4,001	7%
$1,250	$23,315	15%
$1,375	$46,858	24%
$1,500	$69,360	32%

1.14 Interpretation and Conclusions

The results of the PEA considering mining both indicated and inferred mineral resources shows positive economic results for Brewery Creek Project. The PEA has also highlighted the following potential project risks and opportunities:

1.14.1 Project Risks

- Several of the mineral resource areas are characterized by mineralization hosted in relatively thin intrusive sills that are in immediate contact with highly preg robbing carbonaceous sediments. Depending upon the degree of practical mining selectivity, a portion of the resource may be lost due to mining constraints;
- Risks also pertain to the current selection of waste rock storage areas, which have not been thoroughly investigated in terms of environmental aspects;
- Though this report indicates a positive NPV at 5%, Tetra Tech EBA has found the project very sensitive to gold price and operating cost, a 20% unfavourable change could render the project uneconomical;
- The climatic conditions pose additional risk for the proposed operation. Operations in sub-arctic conditions require special considerations in terms of operational efficiency of personnel and equipment. The Brewery Creek Property, as proposed, operates as a part-time process, ceasing mining and active crushing during the cold part of the season and assumes an operating season commencing in March and ending in late October; and
- As described in this study, it is proposed to place ore on the leach pad using trucking. The trucks will impart additional compaction to the crushed ore pad. This compaction is a common problem with leach pads, which causes ponding, internal hydrostatic pressure build-up in buried lifts, and blinding. The effect of compaction from truck stacking on the Brewery Creek materials is unknown at this time.

1.14.2 Project Opportunities

- The Brewery Creek area contains numerous shallow mineralized gold systems. To date, no high-grade feeder zones or large mineralized masses have been identified in the project area. Drilling in the Lone Star area in 2012 resulted in the recognition of a potential higher-grade zone of skarn-style mineralization. Gold grades for the currently identified deposits tend to be relatively low so the potential for higher grades in the Lone Star area could help future project economics;

- This PEA considers a mining season from March to October each operating year. It may be possible to place additional material during the off season on the leach pad, thereby increasing annual tonnage processed, though with a lag of 4 maximum 4 months from placement to recovery. This could increase ounces recovered each year; and
- There may be opportunities to renegotiate the payment of some of the royalties to reduce initial operating costs.

1.15 Recommendations

The following recommendations are made to enhance with respect to the Brewery Creek Property:

1. Complete remaining infill drilling on all deposits included in the PEA to increase confidence of Inferred Resources to an Indicated level, and conduct confirmatory drilling at Kokanee, Golden and Lucky to validate historical results for these areas ($1M),
2. Continue with metallurgical and processing test work for the existing deposits and initiate test work for the Classic and Lonestar deposits, initiate metallurgical test work at Kokanee, Golden and Lucky ($500k),
3. Continue with Executive Committee Project Proposal document and associated site investigation and surveys along with the development of a Mine Closure and Reclamation Plan ($750k),
4. Review the potential success of Fort Knox, Alaska operation as a year around operation. This operation is located in similar climate and has just started to test year around mining and leaching. If possible, this may have a significant positive impact on project economics,
5. Conduct a trade-off study to test effect of various production rates on initial capital requirements,
6. Commence assessment of Classic and Lonestar deposits, and increase confidence of current resource; conduct trade-offs to assess viability of these areas as integrated or a stand-alone operations on the property.

An estimated budget of $2.25M is anticipated to be required to fulfil items 1 through 3. Budgets have not been estimated for items 4 through 6 as these may be conducted as internal exercises and may have indirect value added to the current PEA.

The above recommendations should be attempted before a Feasibility level study and associated detailed site work is considered.

2.0 INTRODUCTION AND TERMS OF REFERENCE

Golden Predator Exploration Ltd. (Golden Predator) is a Canadian exploration company listed on the TSX Venture exchange as GPY, with main office located in Edmonton, Alberta. The company is evaluating the economics of mining their wholly owned Brewery Creek Property located in Yukon, Canada, through conventional open pit mining and a heap leaching operation as well as re-processing previously leached material for gold doré recovery on an annual seasonal basis.

In August 2012, Golden Predator commissioned a team of engineering consultants to complete a Preliminary Economic Assessment study (PEA) in accordance with National Instrument 43-101 (NI 43-101). The contributors to the PEA include Tetra Tech EBA Inc. (Tetra Tech EBA) of Vancouver, BC, Tetra Tech Inc. of Tucson, AZ, SGS - E&S Engineering Solutions Inc. (SGS) of Tucson, AZ, Resource Modeling Inc. (RMI) of Stites, Idaho, Gustavson Associates LLC. of Lakewood, Colorado, and Access Consulting Group (Access) of Whitehorse, YT. The report is based on exploration trenching and drilling results, certified laboratory analysis of samples and subsequent analysis of the results.

Tetra Tech EBA has undertaken to compile the study results completed by the various contributions into a document which follows the format prescribed in accordance with NI 43 101 at the level of PEA.

Tetra Tech EBA followed industry best practices in preparing the contents of this report. Data used in this report has been verified where possible and Tetra Tech EBA has no reason to believe that the data was not collected in a professional manner. Technical data provided by Golden Predator for use by Tetra Tech EBA and other consultants in this study is the result of work conducted, supervised, and/or verified by Golden Predator professional staff or their consultants.

The Brewery Creek Property is a historical mining site, in which mining and gold recovery was undertaken between 1996 and 2002 by Viceroy Minerals Corporation (Viceroy).

The PEA considers a production scenario whereby two thirds of the production will be sourced from open pit mining of mineral resources and one third from reprocessing of the material on the existing heap leach. Mineral resources for the East Big Rock, West Big Rock, Lower Fosters, Kokanee, Golden, Lucky, Bohemian and Schooner are included in the PEA. Studies have also been undertaken on the spent ore on the heap leach facility showing the potential for additional gold recovery through crushing and replacement on the heap leach facility.

2.1 Report Authors and Quality Control

This report has been completed by the following Independent Qualified Persons (QP), as defined in NI 43-101:

Name	Designation	Title and Company
Mark Horan	P.Eng.	Senior Mining Engineer, Tetra Tech EBA Inc.
John Holley	P.E.	Principal Metallurgical Engineer, SGS - E&S Engineering Solutions Inc.
Donald Hulse	P.E.	Principal Mining Engineer, Gustavson Associates LLC
Claiborne Newton, III	Ph.D., SME(RM)	Chief Geologist, Gustavson Associates LLC
Mike Lechner	P.Geo.	President, Resource Modeling Inc.
James Barr	P.Geo.	Senior Geologist, Tetra Tech EBA Inc.
Marvin Silva	Ph.D., P.Eng.	Senior Geotechnical Engineer, Tetra Tech EBA Inc.

The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to the authors at the time of preparation of this report;

- Assumptions, conditions, and qualifications as set forth in this report; and
- Data, reports, and other information supplied by Golden Predator and other third party sources.

2.2 Site Visits

Site visits to the property undertaken by reporting QPs are listed in the table below:

Table 2-1: Date of Qualified Person's Site Visit

Author	Date of Site Visit
Mark Horan	October 16 - 18, 2012
Joe Keane	September 12 - 13, 2012
Donald Hulse	n/a
Claiborne Newton	June 4 - 5, 2013
Mike Lechner	October 16 - 18, 2012
James Barr	March 19 - 21, and May 30 - 31, 2012
Marvin Silva	n/a

2.3 Effective Date of Technical Report

The Effective Date of this Technical Report is July 22, 2014. This date has been selected as the date on which all sources of data were received for inclusion to the technical studies and preliminary economic assessment documented herein.

- The mineral resources contained in the report were previously documented in a Technical Report prepared by Gustavson (Effective date: June 1, 2013, Issued: October 23, 2013) which includes drilling data up to and including hole BC12-559 (November 15, 2012).
- Final laboratory metallurgical results were issued by McClelland Laboratories Inc. (MLI) on July 3, 2013.
- Interim pit designs, mine plan and waste scheduling were issued by Tetra Tech EBA on May 6, 2014.
- Final Heap Leach Facility infrastructure design for cells 8-10, water balance and stacking plan was issued by Tetra Tech Inc. on July 22, 2014.

3.0 RELIANCE ON OTHER EXPERTS

Information regarding ownership of the property, royalties and status of permits has been provided to Tetra Tech EBA by Golden Predator. This information has not been legally verified by Tetra Tech EBA but is believed to be reliable based on review of publically available information. Information has also been obtained through Golden Predator from BBA Engineering in respect of a proposed utility power line, Tetra Tech EBA has read the conceptual study but has not independently verified this information.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Brewery Creek Property includes a past producing heap leach gold mining operation, located in north western Yukon, approximately 55 km due east of Dawson City. The property is centered at Latitude 64.041887° N and Longitude 138.206389° W or UTM NAD83 Zone 7N at 636,401(m) E; 7,104,673(m) N.

4.1 Locality Plan/Map

Figure 4–1 shows the location of the Brewery Creek Property, shown in red, in relation to Dawson City, the Klondike Highway and the Dempster Highway.

4.2 Mineral Titles

The Property consists of total 1,075 quartz claims (93 of which have been converted to quartz leases) covering approximately 181 km² as shown in Figure 4–2. Quartz Claims and licences registered to Golden Predator are listed in Table 4-1.

Tetra Tech EBA has not independently verified the legal status or title of the claims or exploration permits, and has not investigated the legality of any of the underlying agreement(s) that may exist concerning the Property.

Figure 4–1: Locality Map for Brewery Creek, Dawson City, Yukon Territory

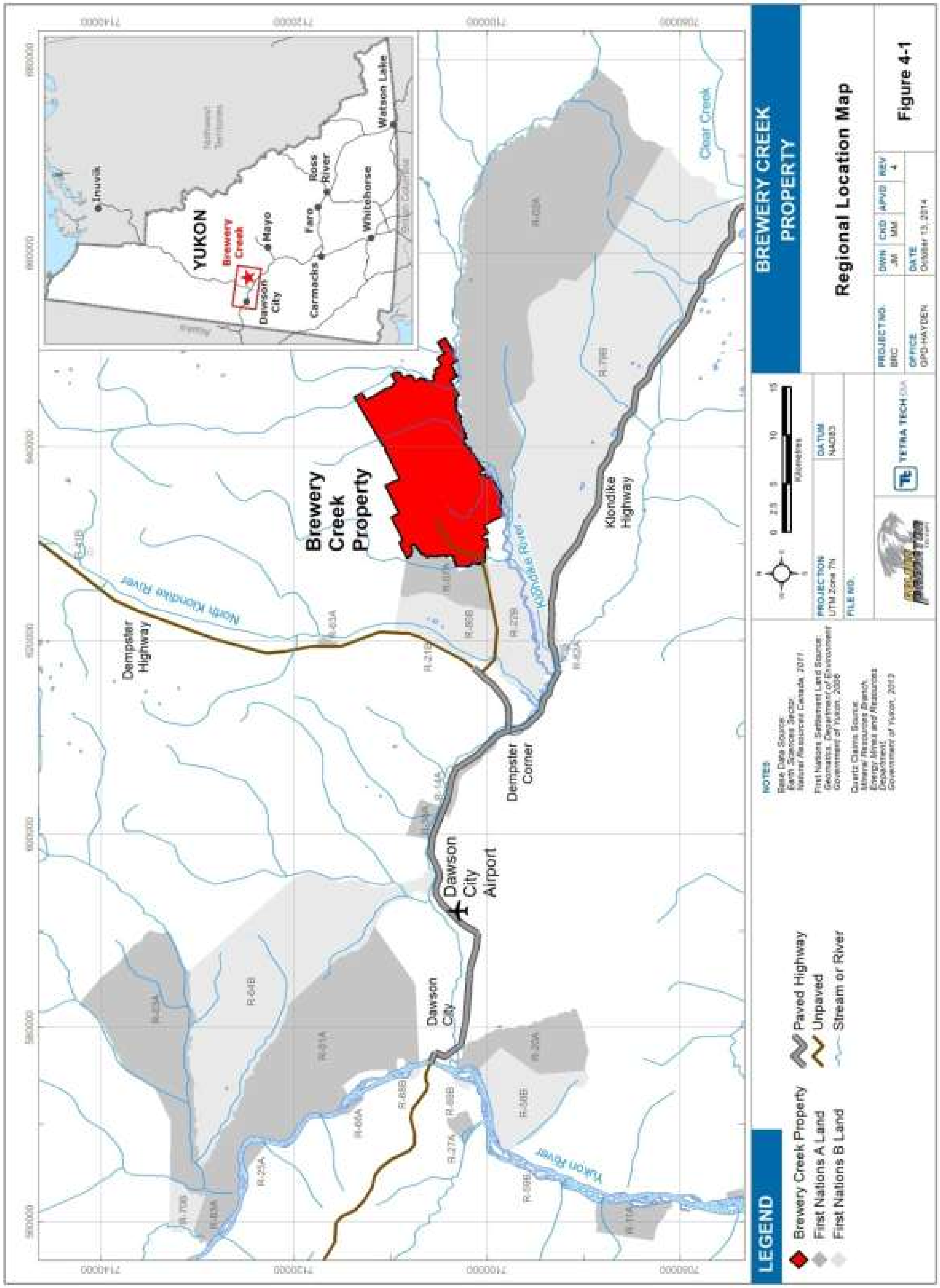


Figure 4–2: Quartz Claims and Quartz Mining Leases

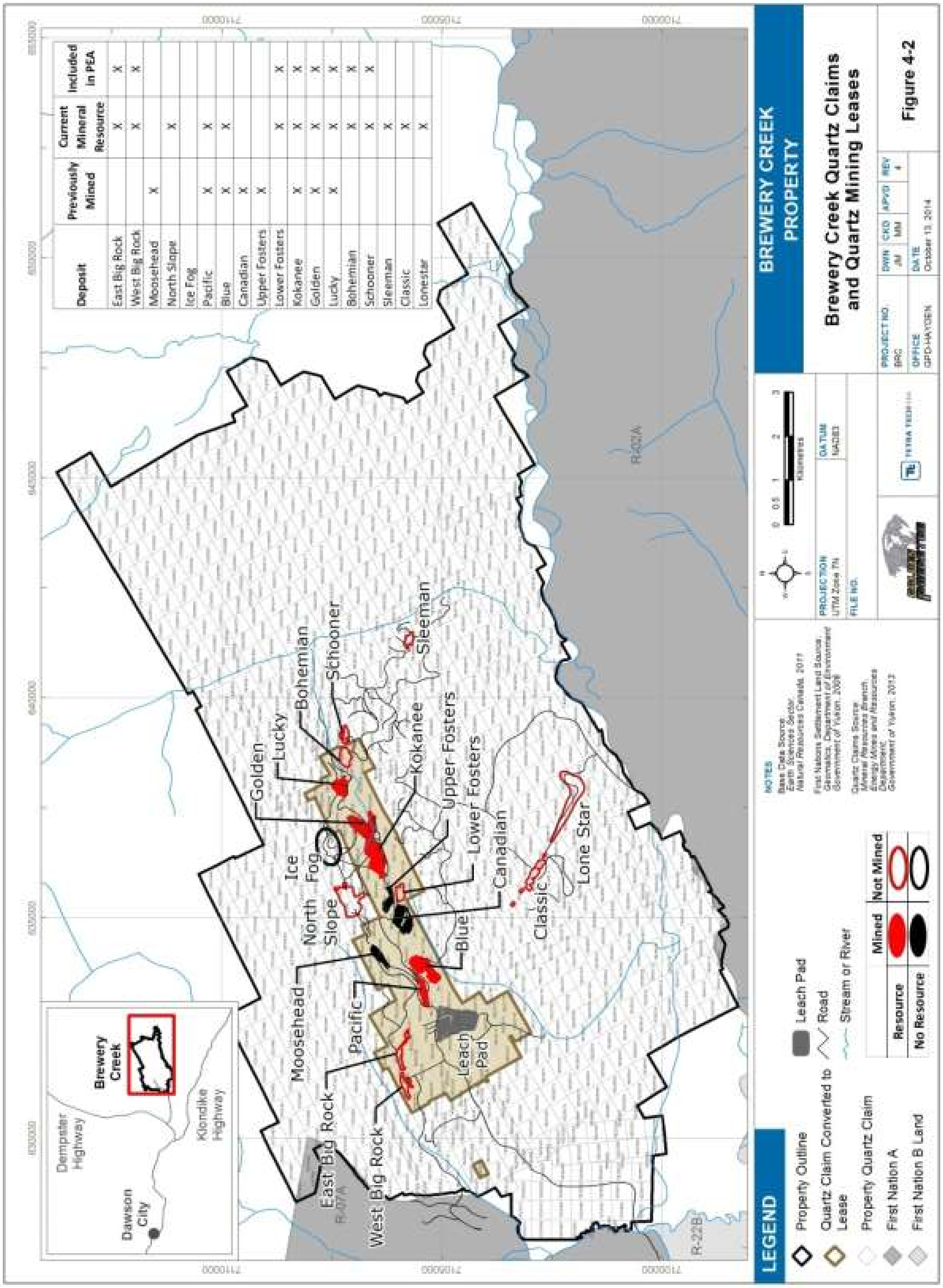


Figure 4–3: Quartz Claims and Property Royalty Agreements

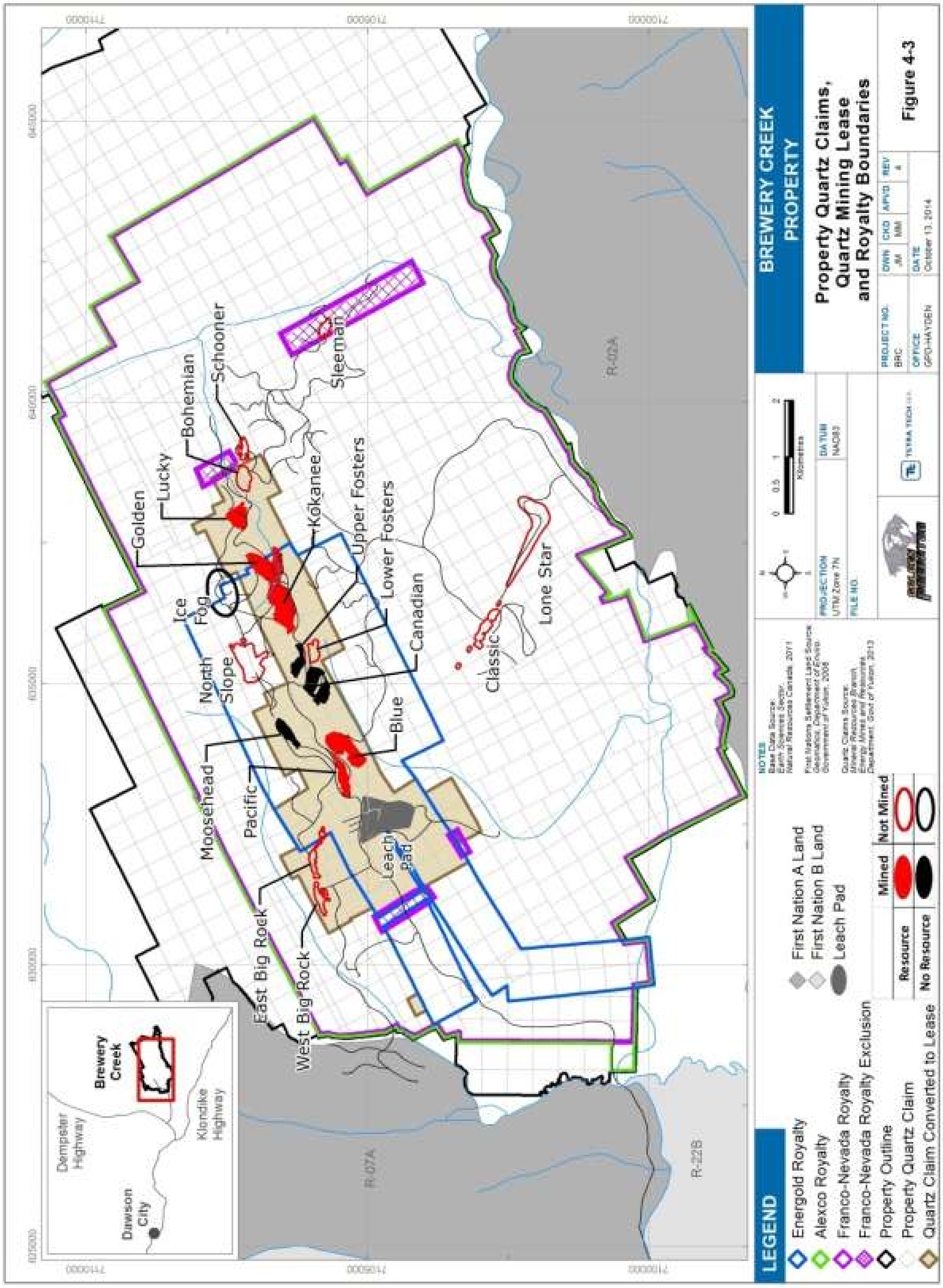


Table 4-1: Brewery Creek Quartz Claims and Licenses

Claim Name	Claim Number(s)	Grant Number(s)	Registered Owner	Expiry Date
BCX	1-2	YD102641-YD102642	Golden Predator Canada Corp. - 100%	5/13/2017
BCX	4-6	YD86503-YD86506	Golden Predator Canada Corp. - 100%	5/13/2017
BCX	9-50	YD86509-YD86550	Golden Predator Canada Corp. - 100%	5/13/2017
BCX	53-131	YD86553-YD86631	Golden Predator Canada Corp. - 100%	5/13/2017
BCX	134-204	YD86634-YD86704	Golden Predator Canada Corp. - 100%	5/13/2017
BDM	1	YB52721	Golden Predator Canada Corp. - 100%	1/20/2019
BDM	2-8	YB52881-YB88626	Golden Predator Canada Corp. - 100%	3/24/2018
Eel	53	YB23907	Golden Predator Canada Corp. - 100%	4/30/2019
Eel	54	YB23908	Golden Predator Canada Corp. - 100%	1/20/2025
Eel	55	YB23909	Golden Predator Canada Corp. - 100%	4/30/2019
Eel	56	YB23910	Golden Predator Canada Corp. - 100%	1/20/2025
Eel	57	YB23911	Golden Predator Canada Corp. - 100%	4/30/2019
Eel	58-66	YB23912-YB23920	Golden Predator Canada Corp. - 100%	1/20/2025
Eel	67-115	YB39516-YB39564	Golden Predator Canada Corp. - 100%	1/20/2022
Eel	116-274	YB39565-YB39721	Golden Predator Canada Corp. - 100%	5/31/2016
Eel	275-277	YB40246-YB40248	Golden Predator Canada Corp. - 100%	4/30/2019
Eel	278	YB40249	Golden Predator Canada Corp. - 100%	5/31/2016
Eel	279	YB40250	Golden Predator Canada Corp. - 100%	3/24/2018
Eel	280	YB40251	Golden Predator Canada Corp. - 100%	5/31/2016
Eel	281-288	YB40252-YB40259	Golden Predator Canada Corp. - 100%	1/20/2023
Eel	289-292	YB40260-YB40263	Golden Predator Canada Corp. - 100%	5/31/2016
Eel	296-297	YB40264-YB40268	Golden Predator Canada Corp. - 100%	1/20/2023
Eel	298	YB40269	Golden Predator Canada Corp. - 100%	5/31/2016
Eel	301-307	YB40283-YB40288	Golden Predator Canada Corp. - 100%	1/20/2023
Eel	299-300	YB40321-YB40322	Golden Predator Canada Corp. - 100%	1/20/2023
Eel	304	YB40323	Golden Predator Canada Corp. - 100%	1/20/2023
Eel	313-352	YB40326-YB40365	Golden Predator Canada Corp. - 100%	1/20/2019
Eel	308-312	YB40366-YB40370	Golden Predator Canada Corp. - 100%	1/20/2023
Eel	313-464	YB40371-YB40482	Golden Predator Canada Corp. - 100%	1/21/2023
Eel	415A-416A	YB40485-YB40486	Golden Predator Canada Corp. - 100%	1/20/2020
Eel	465-478	YB40557-YB40570	Golden Predator Canada Corp. - 100%	1/20/2023
Eel	407A-408A	YB40483-YB40484	Golden Predator Canada Corp. - 100%	1/20/2020
EEL #	1	YB23313	Golden Predator Canada Corp. - 100%	1/20/2025
EEL #	2-10	YB23314-YB23322	Golden Predator Canada Corp. - 100%	5/31/2016
EEL #	11-20	YB23323-YB23332	Golden Predator Canada Corp. - 100%	1/20/2025
EEL #	21-31	YB23333-YB23342	Golden Predator Canada Corp. - 100%	5/31/2016
EEL #	31-52	YB23343-YB23364	Golden Predator Canada Corp. - 100%	1/20/2025
Eel F	465-470	YB45736-YB45741	Golden Predator Canada Corp. - 100%	1/20/2021

Table 4-1: Brewery Creek Quartz Claims and Licenses

Claim Name	Claim Number(s)	Grant Number(s)	Registered Owner	Expiry Date
EELX	1-78	YD03401-YD03478	Golden Predator Canada Corp. - 100%	12/19/2016
Ele	1-4	YB23541-YB23544	Golden Predator Canada Corp. - 100%	1/20/2025
Ele	5-8	YB23545-YB23548	Golden Predator Canada Corp. - 100%	5/31/2016
Ele	9	YB23549	Golden Predator Canada Corp. - 100%	3/24/2022
Ele	10	YB23550	Golden Predator Canada Corp. - 100%	5/31/2016
Ele	11-16	YB23551-YB23556	Golden Predator Canada Corp. - 100%	1/20/2025
Ele	17-20	YB23773-YB23776	Golden Predator Canada Corp. - 100%	5/31/2016
Ele	21-80	YB23777-YB23836	Golden Predator Canada Corp. - 100%	1/20/2025
F/BCX	7-8	YD86507-YD86508	Golden Predator Canada Corp. - 100%	5/13/2017
F/BCX	51-52	YD86551-YD86552	Golden Predator Canada Corp. - 100%	5/13/2017
F/BCX	132-133	YD86632-YD86633	Golden Predator Canada Corp. - 100%	5/13/2017
Flee	1-32	YB23923-YB23954	Golden Predator Canada Corp. - 100%	1/20/2025
Flee	35	YB23957	Golden Predator Canada Corp. - 100%	3/24/2022
Flee	36	YB23958	Golden Predator Canada Corp. - 100%	5/31/2016
Flee	37	YB23959	Golden Predator Canada Corp. - 100%	1/20/2025
Flee	38	YB23960	Golden Predator Canada Corp. - 100%	5/31/2016
Flee	39-50	YB23961-YB23972	Golden Predator Canada Corp. - 100%	1/20/2025
Flee	51	YB23973	Golden Predator Canada Corp. - 100%	5/31/2016
Flee	52	YB23974	Golden Predator Canada Corp. - 100%	1/20/2025
Flee	53	YB23975	Golden Predator Canada Corp. - 100%	5/31/2016
Flee	54-85	YB23976-YB38731	Golden Predator Canada Corp. - 100%	1/20/2025
Flee	105-117	YB40270-YB40282	Golden Predator Canada Corp. - 100%	1/20/2023
Flee	118-121	YB40317-YB40320	Golden Predator Canada Corp. - 100%	1/20/2023
FLEE F	91-95	YB40131-YB40135	Golden Predator Canada Corp. - 100%	1/20/2023
FLEE F	96-97	YB40136-YB40137	Golden Predator Canada Corp. - 100%	5/31/2016
FLEE F	98-104	YB40139-YB40145	Golden Predator Canada Corp. - 100%	1/20/2023
Lee	1	YB04486	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	2	YB04487	Golden Predator Canada Corp. - 100%	1/20/2026
Lee	3	YB04488	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	4	YB04489	Golden Predator Canada Corp. - 100%	1/20/2026
Lee	5	YB04490	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	6	YB04491	Golden Predator Canada Corp. - 100%	1/20/2026
Lee	7	YB04492	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	8	YB04493	Golden Predator Canada Corp. - 100%	1/20/2026
Lee	9	YB04494	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	10	YB04495	Golden Predator Canada Corp. - 100%	1/20/2026
Lee	11-16	YB04496-YB04501	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	17-26	YB04502-YB04511	Golden Predator Canada Corp. - 100%	1/20/2026

Table 4-1: Brewery Creek Quartz Claims and Licenses

Claim Name	Claim Number(s)	Grant Number(s)	Registered Owner	Expiry Date
Lee	27	YB04512	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	28	YB04513	Golden Predator Canada Corp. - 100%	1/20/2026
Lee	29	YB04514	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	30	YB04515	Golden Predator Canada Corp. - 100%	1/20/2026
Lee	31	YB04516	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	32	YB04517	Golden Predator Canada Corp. - 100%	1/20/2026
Lee	33-36	YB17700 - YB17703	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	37	YB17704	Golden Predator Canada Corp. - 100%	1/20/2027
Lee	38	YB17705	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	39	YB17706	Golden Predator Canada Corp. - 100%	1/20/2027
Lee	40	YB17707	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	41	YB17708	Golden Predator Canada Corp. - 100%	1/20/2027
Lee	42	YB17709	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	43	YB17710	Golden Predator Canada Corp. - 100%	1/20/2027
Lee	44	YB17711	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	45	YB17712	Golden Predator Canada Corp. - 100%	1/20/2027
Lee	46	YB17713	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	47	YB17714	Golden Predator Canada Corp. - 100%	3/24/2022
Lee	48-56	YB17715-YB17723	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	57-76	YB17724-YB17743	Golden Predator Canada Corp. - 100%	1/20/2027
Lee	77-78	YB23207-YB23208	Golden Predator Canada Corp. - 100%	1/20/2025
Lee	79-82	YB23209-YB23212	Golden Predator Canada Corp. - 100%	4/30/2023
Lee	86-87	YB38732-YB38733	Golden Predator Canada Corp. - 100%	5/31/2016
Lee	88-89	YB40324-YB40325	Golden Predator Canada Corp. - 100%	1/20/2023

4.3 Royalties and Agreements

Golden Predator Exploration Ltd. currently owns the right to the Brewery Creek Property through a series of transactions and corporate restructuring from Golden Predator Corp (February, 2012), to Golden Predator Canada Corp a wholly owned subsidiary of America's Bullion Royalty Corp (announced January, 2013), to Northern Tiger Resources Inc., to Golden Predator Exploration Ltd. (TSE: GPY).

4.3.1 Property Purchase Agreement with Alexco Resource Corp.

In February 2012, Golden Predator Corp. signed a Purchase Agreement with Alexco Resource Corp. (Alexco) whereby Golden Predator Corp. would acquire a 100% interest in the Brewery Creek Property subject to a 2% NSR in favour of Alexco. The Purchase Agreement was closed in September 2012.

In exchange for a 100% interest in the Brewery Creek Property and in addition to the NSR, the Company paid CAD $3,205,000 to Alexco, representing the cash consideration to be paid under the Purchase Agreement (CAD $4,000,000) less the amount of the reclamation bond that had been posted by Alexco with the Yukon government

(CAD $795,000). The current quartz mining licence and water licence have been transferred to Golden Predator from Alexco.

4.3.2 Royalties

The Brewery Creek Mine is also subject to several generations of underlying Royalties, including the Alexco Royalty, the Franco-Nevada Royalty and the Energold Royalty.

Tetra Tech EBA has reviewed the royalty agreements, and believes the information presented to be true, however, has not sought independent legal advice to verify the details and validity of the agreements.

4.3.2.1 Alexco Royalty

As per the Purchase Agreement dated February 14, 2012, Alexco will retain a 2% Net Smelter Return Royalty (“NSR”) on the next 600,000 ounces of gold produced from the 793 claims acquired from Alexco (Figure 4–3), following which the royalty will increase to 2.75%. Golden Predator has the right to repurchase 0.625% of the increased royalty by paying Alexco $2,000,000.

4.3.2.2 Franco-Nevada Royalty

As per the Royalty Agreement dated September 24, 1993, Loki Gold Corporation (Loki) agreed to pay a US$10 to US$40 per ounce Sliding Scale Royalty (SSR) for 300,000 ounces of gold production in favour of Newmont Canada Limited (a successor to Hemlo Gold Mines Inc.) on 135 claims (Figure 4–3). In 2005, interests in the Brewery Project, including the Newmont Royalty obligation, were transferred to Alexco Corporation from Loki. In 2007, as per an agreement dated December 20, 2007, Newmont Canada Limited assigned the above Royalty benefit to Franco-Nevada Corporation. In 2012, as per the Purchase Agreement dated February 14, 2012, Golden Predator signed an agreement to acquire a 100% interest in the Brewery Creek Property including the Franco-Nevada Royalty obligations from Alexco.

There has been approximately 279,000 ounces of gold production reported for the property, thereby allowing for 21,516 ounces of gold remaining as part of the Royalty Agreement.

The amount of SSR to be paid is based on the price of gold, as follows:

- $10/oz if the Average Gold Price is $349.99 per ounce gold, or less;
- $20/oz if the Average Gold Price is greater than $349.99 and equal to or less than $399.99 per ounce of gold;
- $30/oz if the Average Gold Price is greater than $399.99 and equal to or less than $449.99 per ounce of gold;
- $40/oz if the Average Gold Price is greater than $449.99 per ounce of gold.

4.3.2.3 Energold Royalty

In 1989, as per the Royalty Acknowledgment Agreement dated June 1989, Noranda Exploration Company Limited (Norex) agreed to pay a 5% Net Profits Royalty (NPR) to Total Erickson Resources Ltd. (TERL) on 781 claims (Figure 4–3). In 1992, as per the Agreement dated September 23, 1992, Norex assigned the Royalty obligation to Hemlo Gold Mines Inc. and in 2005, as per the agreement dated March 15, 2005; TERL assigned all of its interest in the Royalty to Energold Minerals Inc. (“Energold”). In 1993, Loki Gold Corporation acquired all of Hemlo’s right title and the Royalty obligation. In 2005, as per the Sale and Assignment Agreement dated February 1, 2005, Viceroy Minerals Corporation (Formerly Loki Gold Corp.) sold its interests in the Brewery Property including the Royalty to Alexco Resources Corp. In 2012, as per the Purchase Agreement dated February 14, 2012, Golden

Predator signed a purchase agreement to acquire a 100% interest in the Brewery Creek Property including the Royalty obligations to Energold from Alexco.

4.3.2.4 Till Capital Royalty

Upon transfer of the Brewery Creek Property from Resource Re Ltd. (subsidiary of Till Capital, and successor of Americas Bullion Royalty Corp.) to Golden Predator Exploration Ltd., a Net Smelter Return Royalty equal to 0.5% was established payable to Resource Re Ltd. applicable to the final settlement amount paid from the smelter, refinery or other processing facility to which concentrate or other product derived from the Brewery Creek Project were sold.

4.3.2.5 Permits and Status

Brewery Creek holds a Quartz Mining License (QML A99-001) for the production of minerals pursuant to the Yukon Quartz Mining Act. This license has an expiry date of December 31, 2021. Name transfer from Alexco to Golden Predator was completed during 2012.

Brewery Creek is authorized under a Type A Water Use License (QZ96-007) to obtain and use up to 2,724 m³ of water per day from Laura Creek (a tributary of the South Klondike River), and to deposit waste, as defined in Viceroy Minerals Corporation’s water license application, into the catchment basins of Laura, Lucky and Pacific Creeks. The expiry date of the Water License is December 31, 2021.

Brewery Creek is also authorised under a type B Water Use Licence (MN12-038) to obtain 50 m³ per day for the camp and to dispose in the approved onsite septic system.

Current exploration at Brewery Creek is conducted under an active Class 4 Mining Land Use Permit (LQ00364), which expires on July 5, 2022. Final reclamation, including re-establishment of vegetative mat and erosion control, must be completed according to the terms and conditions of the permit prior to the expiry date.

By acquisition of the Property, Tetra Tech EBA believes that the authorizations and licences listed above have been transferred to Golden Predator from Alexco; however, Tetra Tech EBA has not sought independent legal advice to verify this.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Property is located approximately 55 km east of Dawson City, YT and is accessible by paved and gravel road. To access the site from Dawson City, YT drive 40 km east on the paved Klondike Highway; 8 km north on the all-weather Dempster Highway; then eastward for 20 km on the upgraded North Fork Road to the south western edge of the property, and finally another 6 km to the mine site, on a company maintained road (Figure 4–1).

Located at approximately 64° north latitude, the Property is subject to a subarctic climate with average temperatures ranging from 15°C (60°F) in July to -26°C (-16°F) in January with temperatures commonly reaching above 30°C (86°F) in the summer and below -40°C (-40°F) in the winter. Average annual precipitation at Brewery Creek is approximately 325 mm and annual frost free days is approximately 110 days.

The Property is located in the Ogilvie Mountains immediately north of the Klondike River and west of Lee Creek. Elevations on the Property range from approximately 450 metres (1,500 feet) to 1,200 metres (4,000 feet). Relief on the property varies from moderately steep in the southwest corner of the property to steep and very steep for the majority of the Property. The area was not glaciated during the last glaciation period resulting in relatively steep

V-shaped valleys incised by the creeks that cross the property. Natural bedrock exposure is generally less than 1% and is restricted to the higher elevation ridges within the Property area.

Vegetation on the Property consists of four main types. The higher elevations (above 1050 metres) consist of rounded hills covered with sub-alpine shrubs, grasses and widely spaced coniferous trees. Steep north facing slopes and narrow valley floors are covered with thick blankets of moss with thickets of slope alder and stunted spruce. Steep south facing slopes have two distinct styles of vegetation; coniferous trees with abundant undergrowth and areas of deciduous aspen, poplar and birch with little or no undergrowth.

Tetra Tech EBA undertook a surficial terrain study on the Brewery Creek Property in 2012. The results of the study show that all north facing slopes and valley bottoms are influenced by permafrost. Areas of gentle topography, especially NW facing slopes, and gullies contain loess (fine wind-blown silt) up to 17 metres thick. Observed geomorphological processes include slow soil creep on the middle to lower slopes of some stream valleys and minor sloughing along some eroded stream banks. There were no indications of active rapid mass movement processes observed during the field visit. Some minor sloughing on fill slopes of existing waste piles appears to have occurred in the past, but these do not appear to be active. Some sloughing and sliding of a minor volume of organic-rich overburden at a new exploration road and drill site in the Bohemian-Schooner area was reported. The majority of the proposed mining and operational area are judged to be free of permafrost; however, subsurface data to confirm permafrost is limited. Permafrost in the study area is discontinuous and is probable on most lower slopes and floors of the moderately steep (50% to 70% gradient) V-shaped stream valleys.

5.1 Infrastructure

The original administration building, with an office, core logging facility, warehouse and storage still exists on site along with the heap leach pad, process and overflow ponds from the original mine. This infrastructure has been on site since 1996 Viceroy commenced operations at the mine. In 2011, Golden Predator added several mobile accommodation trailers and wall tents to accommodate an increase in staff and personnel, and upgraded the site roads for navigation and mobilization of exploration equipment.

5.1.1 Existing Roads

The Brewery Creek Property is accessible year round by gravel road. The access road is in good condition and is maintained partially by the Yukon government and partially by Golden Predator. Two bridge crossings that were built for initial Viceroy construction and closure phases remain active and visually in good repair. No structural inspections have been completed on these structures.

A network of site roads exist that connect various exploration sites and work locations to the main camp. Previously reclaimed roads have been brought into use and maintained by Golden Predator using on-site rented earth moving equipment.

On-site haul roads were partially rehabilitated and are passable by light vehicle and equipment used for exploratory drilling. Rehabilitation work included re-contouring of haul roads to fit into topography, placement of topsoil and removal of culverts over streams.

Table 5-1: Haul Road Lengths and Condition of Road

Road Segment	Approximate Length	Elevation Change	Condition
Camp to Fosters Pit	4.5 km	+60 m	Good condition, with adequate space to increase width of the road for haul road construction, passable by light pickup

Table 5-1: Haul Road Lengths and Condition of Road

Road Segment	Approximate Length	Elevation Change	Condition
Fosters to Bohemian Pit	4.2 km	-40 m	Good road to Lucky pit after which road narrows and steepens to Bohemian pit area, passable with 4 wheel drive
Camp to East Big Rock Pit	3.5 km	- 80 m	Good condition but narrow, passable by light pickup. New haul road position along slope lower down suggested
Camp to West Big Rock Pit	1.7 km	- 80 m	Good condition but narrow, passable by light pickup. New haul road position along slope to the north of the camp suggested
Camp to ADR	1.9 km	- 70 m	Good condition, very minor maintenance required for operational use, passable by light pickup
ADR to Laura Creek Pumphouse	1.8 km	- 110 m	Good condition, minor maintenance required for more frequent use, passable by light pickup

The existing road network will be upgraded for mining purposes. Refer to section 18.14 for details of the access and haul road upgrading requirements.

5.1.2 Existing Laydown Area

A temporary laydown area exists on the Brewery Creek mine site east of the current administration building, with an approximate area of 0.5 ha. This area has been used to store various items used during the current exploration phase of the project. This area will be used for the construction of the truck shop for the operations, and the laydown area may move to the west of the camp where a warehouse will be constructed for spares and equipment for the operations.

5.1.3 Existing Process Ponds

Reclaimed ponds are located to the southwest of the heap leach facility and include the pregnant pond, barren pond, and overflow pond as they are viewed from west to east. The ponds remain excavated and are currently holding water. All piping and liners were removed during reclamation and the silt base has been left in place reducing drainage of water during the years the mine has been reclaimed. During Viceroy operations, an additional pond was constructed north of the ADR facility. The purpose of this pond was not apparent at the time of the study but may be have been a settlement pond for surface drainage.

Table 5-2: Golden Predator Anticipates Rebuilding and Activating the Following Ponds for Production

Pond	Surface area	Status
Pregnant pond	6,148 m²	Needs grading and re-lining
Barren pond	6,103 m²	Needs grading and re-lining
Overflow pond	12,930 m²	Needs grading and re-lining

5.1.4 Existing Heap Leach Facility

Leached ore currently located on the heap leach pad (Figure 5–1) was partially rehabilitated after closure of the Viceroy operations. Following a detoxification process involving the circulation of water through the heap leach to dissolve the cyanide, the material comprising three 10 metre lifts was contoured and shielded from meteoric runoffs through the placement of a topsoil cover. It is estimated that approximately 60% of the contained gold was historically recovered from this material (refer to Section 14).

The facility was designed and permitted for a capacity of approximately 15 million tonnes of ore within 10 cells. Capacity for Cells 1 through 6 has been reached and approximately 1 million tonnes is estimated to remain on Cell 7. The remaining three cells consist of an area of approximately 190,000 m², with an estimated capacity for roughly 5.5 million tonnes.

The leak detection system for the existing liner of cells 1 through 7 is still installed and is assumed to be in working order. Tests have not been conducted to confirm it functions properly.

Golden Predator intends to construct and line the remaining three cells for placement of ore material.

The ADR plant and associated piping was removed from the property during mine closure.

Figure 5–1: Overview of the current reclaimed heap leach facility and solution ponds (Golden Predator, 2012)

6.0 HISTORY OF THE PROPERTY

6.1 Ownership

The initial Project claims were staked by Noranda Exploration (Norex) in 1987 to cover a reconnaissance geochemical anomaly. Further claims were staked in subsequent years to cover possible extensions of gold mineralization. In June 1990 Loki Gold Corporation (Loki) entered into an option agreement with Norex and earned a 49% interest in the property by August 1991 by spending $4 million in exploration. In June 1993, the remaining 51% interest was purchased, giving Loki sole ownership of the property.

In 1994, the claims covering the deposit areas, mine facilities and heap leach pad area were surveyed and taken to lease. In May 1996 Loki amalgamated with Baja Gold, Inc. to form a new company under the name VLB Resource Corporation, a wholly-owned subsidiary of Viceroy Resource Corporation. VLB Resource Corporation in turn changed its name to Viceroy Minerals Corporation (Viceroy). Mine commissioning, production, closure and reclamation occurred under Viceroy ownership.

On May 1, 2003, an agreement amongst Viceroy, 650399 BC Ltd., Spectrum Gold Inc. and NovaGold Canada Inc. (NovaGold). was established in which Viceroy would allow 650399 BC Ltd an option to purchase mineral properties of, other rights to, and assets of the Brewery Creek Gold Mine (Diment and Simpson, 2003). At this time, 650399 BC Ltd. (BC) was a wholly owned subsidiary of SpectrumGold Inc. (Spectrum).

A small drilling program was conducted by 650399 BC Ltd. in 2004. Later that year, NovaGold acquired all of the outstanding shares of SpectrumGold and thus the option for assets of the Brewery Creek Gold Mine.

In April 2005, NovaGold relinquished the option for Brewery Creek claims and mining leases to Alexco) with a back-in clause following the completion of $700,000 of exploration expenditures by Alexco. NovaGold elected not to participate with this back-in option.

In 2009, Golden Predator Corp. signed an option agreement with Alexco whereby Golden Predator Corp. had the option to acquire up to 75% interest in 793 quartz claims and mining leases covering 127 km² by exercising three stages of an Option Agreement. A Purchase Agreement was signed between Golden Predator Corp. and Alexco in February of 2012 by which Golden Predator Corp. purchased 100% ownership in the property, subject to some terms, as described in Section 4.3.1.

In early 2013 Golden Predator Corp. (GPD) changed its name to Americas Bullion Royalty Corp. (AMB) and in the process divested the Brewery Creek assets into the subsidiary Golden Predator Canada Corp. (GPCC). Following the reorganization and transfer of shares from of AMB into a new company called Till Capital (TSE: TIL, listed April 24, 2014), the Brewery Creek assets were acquired by Northern Tiger Resources Inc. who subsequently announced name change to Golden Predator Exploration Ltd. (Golden Predator, TSE: GPY) on April 17, 2014. On date of TIL stock listing, Till Capital held 54% of outstanding shares in Golden Predator.

6.2 Exploration

Historical exploration surveys conducted at Brewery Creek between 1988 and 2006 included geological mapping, extensive grid soil sampling, ground and airborne geophysical studies, mechanized surface trenching, and extensive core and reverse-circulation drilling.

6.2.1 Geologic Mapping

Due to rare exposure of the local bedrock, geological mapping on the site has been restricted primarily to trench and road cut exposures. Scree and soil mapping was also utilized outboard from main exploration zones to develop a coherent and regionally consistent geology map.

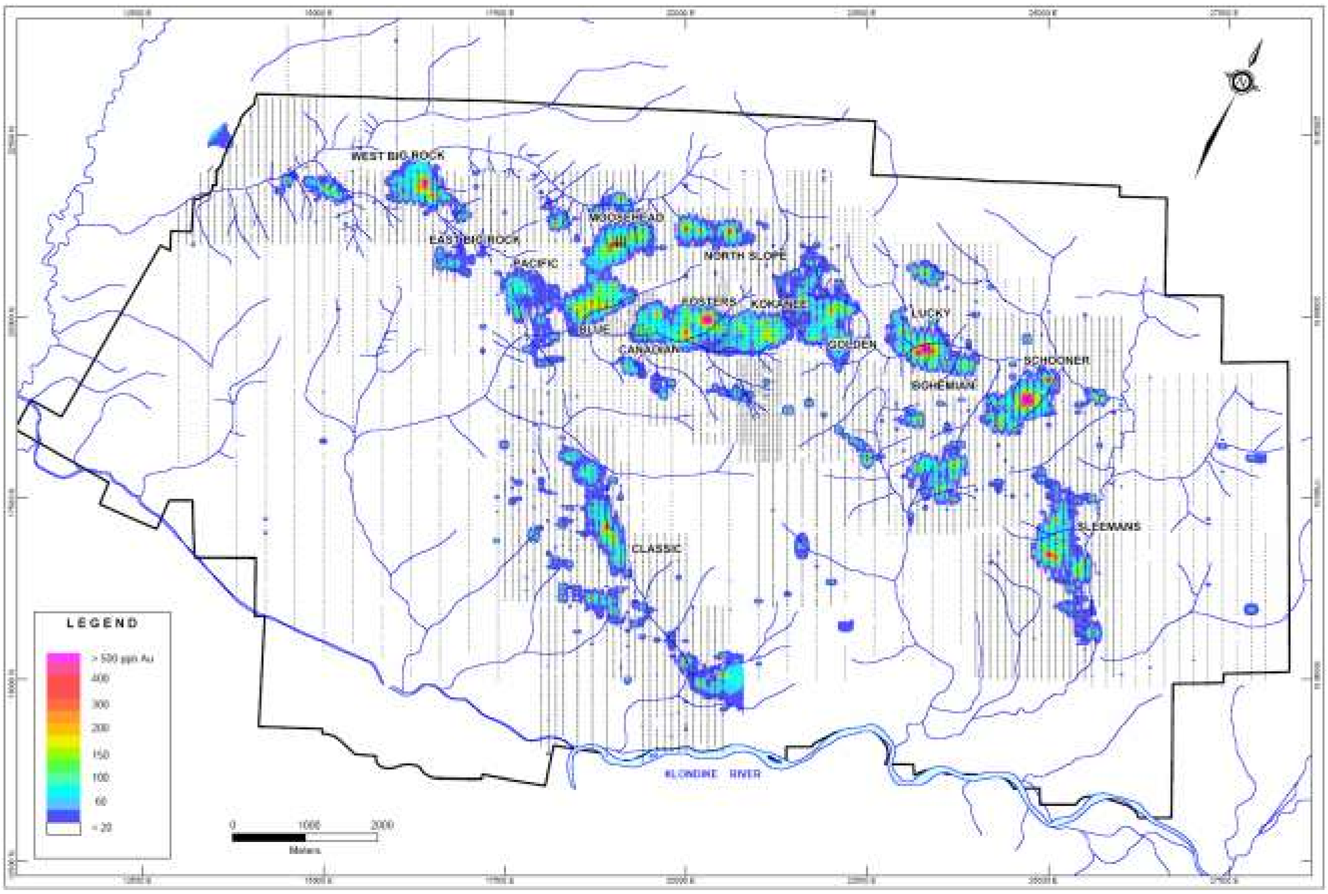
Structural mapping in 2002 by R. Diment and in 2003 by R. Diment and P. Lindberg has developed a more regionally consistent and comprehensive structural model.

A Ph.D. thesis titled The Structural and Hydrothermal Evolution of Intrusion-Related Gold Mineralization at the Brewery Creek Mine, Yukon, Canada, was authored by Mark Lindsay and submitted to the James Cook University, North Queensland, Australia, in May 2006. The work presents a detailed account of mineralogy, alteration and structural implications. The geological mapping is discussed in Section 7.2 of this report.

6.2.2 Soil Sampling Surveys

Soil geochemistry has been an important historical exploration tool and data was collected by most of the previous operators on the property. Gold-in-soil anomalies have assisted in the discovery of all the known mineralized zones and exploration targets (Figure 6–1). Over 24,000 soil samples have been collected on the property to date. The hydrothermal system at Brewery Creek is anomalous in gold, arsenic, antimony and mercury. Silver is weakly anomalous and erratic; it is associated with zinc in the sediments and gold in the epithermal system. Further description of the soil sampling programs is described in Diment and Simpson, 2009.

Figure 6–1: Compilation of Historical Gold in Soil Results (from Diment and Simpson, 2009)



6.2.3 Geophysics

Geophysical surveys consisted of ground magnetometer and IP surveys were conducted between 1989 and 1992 by Norex. In 1998 an airborne magnetometer and radiometric survey was also conducted covering the entire property and adjacent R-7A and R-2A Tr'ondek Hwech'in settlement land. During 2004, 28 km of Induced Polarization (IP) geophysical survey was completed. In 2012 an airborne magnetometer and radiometric survey was conducted covering the entire property.

Although the airborne and ground magnetometer surveys were useful in delineating Tombstone Suite intrusive centers and their adjacent hornfelsed aureoles, mineralized zones typically lie outboard of or flank these magnetic anomalies. The oxidized, auriferous sills that make up most of the Reserve Trend deposits exhibited a relatively flat magnetic response.

Results of the 2004, twenty-eight kilometre IP geophysical survey clearly defined two chargeability domains (west-high and east-low) that are separated by a major northwest trending fault. The trace of this structure passes from the Classic Zone to just west of the Pacific production pit. A strong magnetic-high is coincident with the high chargeability anomaly. Sulfide-bearing intrusive bodies and/or hornfelsed, pyrrhotite-bearing strata may possibly explain the high chargeability features whereas the low chargeability terrain to the east may reflect widespread sulfide destruction linked to the main mineralizing event over the mine trend.

6.2.4 Trenching

Between 1989 and 1999, Loki completed a total of 318 trenches with at cumulative length of 42,300 m were excavated on the property. Further description of the trenching program is described in Diment and Simpson, 2009.

6.2.5 Drilling

A summary of historical drilling conducted from 1989 to 2004 is provided in Table 1-6. Trench and drillhole locations are shown on Figure 6–2.

Table 6-1: Summary of Historical Drilling

Drill Series	Year Drilled	Operator	Drill Type	No. DHs	Total Metres Drilled
RC89	1989	Norex	RC	14	1,704
DD89	1989	Norex	Core	9	1,097
RC90	1990	Loki	RC	309	14,838
DD90	1990	Loki	Core	16	1,090
PQ90	1990	Loki	Core	5	198
RC91	1991	Loki	RC	348	18,007
DD91	1991	Loki	Core	34	1,645
RC92	1992	Loki	RC	19	1,236
RC93	1993	Loki	RC	151	8,542
RC94	1994	Loki	RC	242	10,891
RC95	1995	Loki	RC	317	14,981
DD95	1995	Loki	Core	25	1,200
RC96	1996	Viceroy	RC	271	14,458
DD96	1996	Viceroy	Core	23	2,992
RC97	1997	Viceroy	RC	367	23,045
RC98	1998	Viceroy	RC	219	13,960
DD98	1998	Viceroy	Core	10	662
RC99	1999	Viceroy	RC	53	4,244
BC04	2004	Spectrum	Core	5	770
BC06	2006	Alexco	Core	9	1,171
Total				**2,445**	**136,731**

6.2.6 Norex (1989)

Norex completed 13 reverse circulation (RC) holes, totaling 1,704 metres, near the current Upper Fosters, Canadian, Blue and Kokanee areas, and 9 diamond drillholes, totaling 1,096.8 metres, near the current Upper Fosters, Canadian and Moosehead areas that were completed by Norex in 1989.

The drilling targeted anomalous soil samples and were generally oriented to the north, across dip of geology. Materials intersected in these holes with significant grades have been removed by previous Viceroy mining operations and are not considered to be relevant to the mineral resource estimate presented in this report.

6.2.7 Loki / Viceroy (1990 – 1999)

Golden Predator's drill database has records for a total of 2,296 RC holes drilled between 1989 and 1999 amounting to a total of 124,201.6 metres and a total of 113 core holes drilled between 1989 and 1999 amounting to a total of 7,787.7 metres.

The programs were designed for early exploration and were followed by delineation drilling programs for Viceroy resource and reserve development.

Drilling by Loki and Viceroy was generally conducted in combination of vertical and inclined drilling at 25 to 30 metres spacing along fences offset at 20 to 40 metres across the development areas of interest.

Core recovery was inherently low in many of the core holes due to poor integrity of the wall rock sedimentary rocks. As RC drilling was used as the preferred method for deposit delineation, the limited core drilling post 1989 was restricted to geotechnical drilling for pit wall stability studies, deeper sulfide drilling, and twinning of significant RC hole intercepts for grade and thickness comparisons.

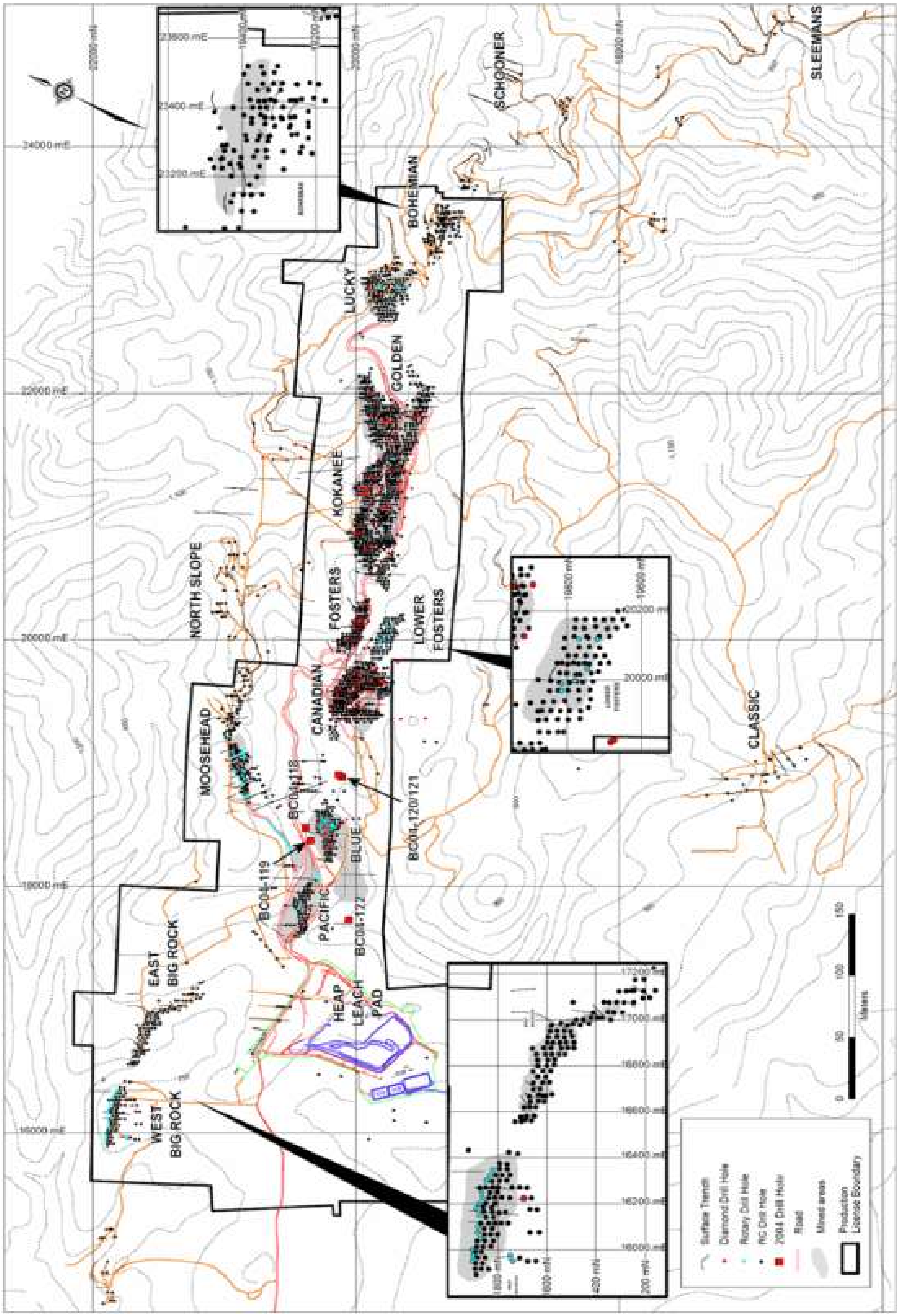
6.2.8 Spectrum (2004)

Following mine closure, core drilling was resumed in 2004 by Spectrum Gold to provide adequate information for structural interpretation during this renewed phase of exploration. Diamond drilling in 2004 tested targets at Blue, Blue East and South Pacific. A total of 5 core holes totaling 770 metres were completed.

6.2.9 Alexco (2006)

Alexco completed a diamond drilling program in 2006, managed by geological personnel from NovaGold. The drill program consisted of 9 HQ core holes for 1,171.53 metres. The drilling was carried out by E. Caron Diamond Drilling Ltd. of Whitehorse, Yukon. Caron supplied the program with two skid mounted Longyear 38 drills with drill pipe sloops, water tank and a water truck, and a D-7 cat for rig moves. The drilling was completed between March 20 and May 1, 2006 at Bohemian, Classic, Blue as well as IP anomalies along a major NW-striking fault extending from the Classic to the Pacific Zone.

Figure 6–2: Trench and Drillhole Locations (Diment and Simpson, 2009)



6.3 Production

The description of the historical production found below has been extracted and modified from Diment and Simpson (2009).

From 1996 through 2002, approximately 279,000 ounces of gold were produced from eight near-surface oxide deposits occurring along strike of the historically termed “Brewery Creek Reserve Trend” (BCRT). A silver credit was included within the doré shipped from site. A Brewery Creek Monthly Report from December 2001 indicates that a total of 276,335 ounces (troy) of gold and 115,574 ounces (troy) of silver had been produced from the project to date.

The first gold pour at Brewery Creek Mine was completed on November 15, 1996 with 10,175 ounces being produced prior to commencement of full commercial production in May of 1997. During 1997, a total of 72,387 ounces of gold were produced at a cash cost of $USD 184 per ounce. In 1998 production totaled 79,396 ounces at a cash cost of $USD 177 per ounce. Production in 1999 fell to 48,164 ounces while operating costs rose to a cash cost of $USD 288 per ounce. Viceroy suspended seasonal mining operations earlier than planned and hired an independent consulting company to study recovery processes in an effort to improve recoveries. In 2000, Viceroy concentrated on selectively mining the mineralized bodies, which were well oxidized and contained the highest grade. Production in 2000 fell to 48,048 ounces of gold at a cash operating cost of $USD 243 per ounce. Mining ceased in 2001, but heap leaching continued with production of 18,542 ounces of gold at a cash operating cost of $USD 222 per ounce.

During 2002, Viceroy undertook and completed approximately 50% of the mine area reclamation related to re-contouring and re-vegetation of pits and dumps. A heap detoxification program was also initiated bringing cyanide and metal levels of heap effluent to water license discharge levels, excluding selenium, by September 2002. An amendment to the water license was approved by government regulatory agencies at this time, allowing land application of heap effluent of up to 200,000 m³ per year. Re-circulation of effluent to the heap ceased in October 2002 excluding 450 l/min that was applied to the heap over the winter (2002/2003) for snow making purposes. A final closure and decommissioning plan was prepared and submitted as required, to the regulatory agencies, and the primary elements of the plan adopted as water license amendments granted in April 2005.

Studies undertaken in the year 2000 on historical heap leach recovery data had shown a recovery of 65% for uncrushed ore. Discussions were raised at the time on the merits of crushing for which studies had shown a potential increase of 10% for the recoveries, at a stated cost of $2.50 per tonne of ore at the time. It should be noted that the recoveries estimated in the preproduction study undertaken in 1995 were 78%.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The northern Cordillera consists of five physiographic domains composed of deformed sedimentary rocks, allochthonous terranes and associated magmatic rocks (Figure 7–1, legend Figure 7–2). From west to east these domains are referred as; Insular, Coast, Intermontane, Omineca and Foreland belts. Within the northern Cordillera, the Tintina Fault generally marks the boundary between the ancient North American craton on the northeast to the allochthonous (accreted) terrains, composed of younger and varying rock types, to the southwest. The Tintina Fault, is interpreted as a Paleogene-aged dextral strike-slip fault with an estimated displacement of at least 450 km, but may be up to 1200 km (Hart, 2011). The fault is marked by the Tintina Trench, a broad valley approximately 15 km wide in the Project area which also extends throughout the Yukon as a the northern extension to the Rocky Mountain Trench. Volcanic rocks were deposited into the Tintina Trench about 55 Ma and it has subsequently filled with young unconsolidated sediments.

Brewery Creek is situated in the Omineca Belt, east of the Tintina Fault in the central northern Cordillera, and is characterized by large mountain ranges and plateaus composed of folded and variably metamorphosed sedimentary and volcanic strata intruded by felsic plutons. The property lies in the foothills of the Ogilvie Mountains, on the northern Stewart Plateau.

Figure 7–1: Regional Geology Map

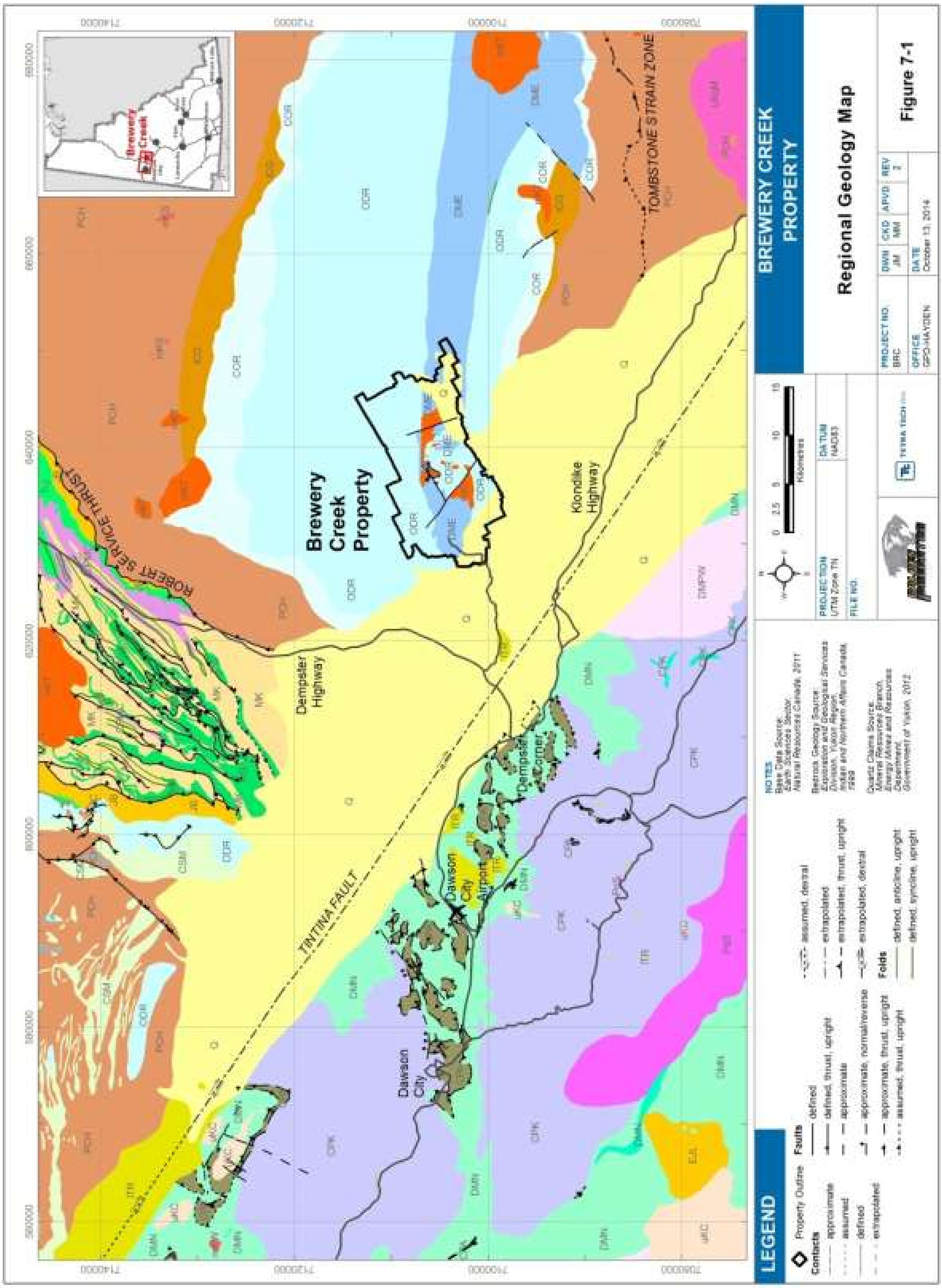


Figure 7–2: Regional Geology Legend

Legend

QUATERNARY

Q: QUATERNARY: unconsolidated glacial, glaciofluvial, glaciolacustrine deposits; fluviatile silt, sand, and gravel, and local volcanic ash, soil and organics

LOWER TERTIARY, MOSTLY(?) EOCENE

lTR: ROSS: brown, thin bedded, claystone, siltstone, shale and coal; arkosic or chert rich, thick bedded micaceous sandstone; to massive pebble to boulder, chert-quartz conglomerate, and light coloured felsic quartz feldspar porphyry and rhyolite; crystal lithic tuff and ignimbrite

LATE CRETACEOUS

LKqM: MCQUESTON SUITE: medium- to coarse-grained, locally porphyritic and K-feldspar megacrystic biotite muscovite granite and quartz monzonite;

MID-CRETACEOUS

mKqT: TOMBSTONE SUITE: medium- to coarse-grained, locally porphyritic biotite hornblende, granite, quartz monzonite and granodiorite (Tombstone Suite)

mKyS: SELWYN SUITE: mainly hornblende and hornblende-biotite syenite, commonly porphyritic (potassium feldspar phenocrysts), uneven textured, mostly medium grained, locally fine or coarse grained; minor diorite; hornblende syenite (Selwyn Suite)

UPPER CRETACEOUS

uKC: CARMACKS: augite olivine basalt; hornblende feldspar porphyry andesite lows; agglomerate and associated epiclastic rocks (Carmacks Gp., Little Ridge Volcanics, Casino Volcanics). Acid crystal, lapilli, and welded tuff, felsic volcanic flows and quartz feldspar porphyries; (Carmacks Gp.,Donjek Volcanics).

JURASSIC

JB: BUG CREEK: grey argillite, slate, and phyllite, commonly graphitic, thin-bedded dark grey quartzite; minor phyllite and limy quartzite (Lower Schist)

MIDDLE TO UPPER TRIASSIC

TrJ: JONES LAKE: brown to buff weathering, calcareous fine grained sandstone, argillite and shale; extensive ripple cross-lamination and bioturbation; massive,light grey weathering, fine crystalline, dark grey limestone; minor orange weathering platy limestone (Jones Lake)

TRIASSIC

TrG: GALENA SUITE: massive, medium-grained hornblende diorite and gabbro sills; massive chloritic and locally serpentinized greenstone (diorite, gabbro; and altered equivalents) sills; minor occurrences of possible mid- to Late Paleozoic age

MIDDLE PERMIAN

PqS: SULPHUR CREEK SUITE: moderately to strongly foliated biotite quartz monzonite gneiss, coarse grained, homogeneous, hornblende-biotite-bearing granite, granodiorite and quartz-monzonite with narrow foliated and mylonitic zones of the Ram Stock (Sulphur Creek Orthogneiss, Ram Stock)

CARBONIFEROUS AND PERMIAN

CPK: KLONDIKE SCHIST: tan to rusty and black weathering muscovitic and/or chloritic quartzite and quartz-muscovite-chlorite schist; quartz and/or feldspar augen-bearing quartz-muscovite (chlorite) schist,augen gneiss and amphibolite (Klondike Schist) and white marble with a ductile flow fabric (Klondike Schist)

LATE DEVONIAN TO MISSISSIPPIAN

DMqPW: PELLY GNEISS SUITE - SOUTHWEST: foliated equigranular medium-grained muscovite quartz monzonite; moderately to strongly foliated K-feldspar augen-bearing quartz monzonitic to granitic gneiss (S. Fiftymile Batholith, Mt. Burnham Orthogneiss.)

DEVONIAN, MISSISSIPPIAN AND(?) OLDER

DMN: NASINA: dark grey to black, fine grained graphitic quartzite, grey micaceous quartzite and quartz muscovite schist, locally garnetiferous; minor graphitic stretched metaconglomerate (Nasina assem.). Marble. Quartzite, quartz muscovite (chlorite; feldspar augen) schist, and minor metaconglomerate

CARBONIFEROUS TO PERMIAN

CPA: ANVIL: altered and foliated, augite-phyric basalt (local pillows), greenstone, and amphibolite; siliceous argillite or siltstone, greywacke, tuff, and limestone. Dunite, peridotite, gabbro, pyroxenite, harzburgite and diorite, hornblendite and diabase; serpentinite, and carbonatized ultramafic rocks

CPMC: MOUNT CHRISTIE: burrowed, interbedded greenish grey cherty shale and green shale; thin to medium bedded, light grey-green to black chert; black siliceous slate and siltstone; minor quartzite, limestone and dolostone; locally abundant, large grey barite nodules (Mount Christie)

MISSISSIPPIAN

MK: KENO HILL: massive to thick bedded quartz arenite; thin to medium bedded quartz arenite interstratified with black shale or carbonaceous phyllite; local scour surfaces and shale intraclasts; locally foliated and lineated (Keno Hill Quartzite)

DEVONIAN AND MISSISSIPPIAN

DME: EARN: thin bedded, laminated slate with thin to thickly interbedded fine to medium grained chert-quartz arenite and wacke; thick members of chert pebble conglomerate; black siliceous siltstone; nodular and bedded barite; rare limestone (Earn Gp., Portrait Lake and Prevost)

ORDOVICIAN TO LOWER DEVONIAN

ODR: ROAD RIVER - SELWYN: black shale and chert overlain by orange siltstone or buff platy limestone; locally contains beds as old as Middle Cambrian; correlations with basinal strata in Richardson Mountains include: ODR with CDR (Road River Gp.)

CAMBRIAN TO SILURIAN

CSM: MARMOT: massive brown to green, basic lapilli tuff, breccias, flows, sills, and dykes; intraclast breccia and conglomerate; brown weathering, green to grey, medium to thick bedded volcaniclastic sandstone, dark grey weathering, massive, locally pillowed, dark grey-green basalt, tuff and breccia (Menzie)

UPPER CAMBRIAN AND ORDOVICIAN

COR: RABBITKETTLE: thin bedded, wavy banded, silty limestone and grey lustrous calcareous phyllite; limestone intraclast breccia and conglomerate; massive to laminated, grey quartzose siltstone and chert and rare black slate; local mafic flows, breccia, and tuff (Rabbitkettle)

LOWER CAMBRIAN

lCG: GULL LAKE: shale, siltstone and mudstone, locally bioturbated, with minor quartz sandstone; rare green-grey chert; local basal limestone and limestone conglomerate; phyllite to quartz-muscovite-biotite schist (garnet sillimanite staurolite andalusite) (Gull Lake)

UPPER PROTEROZOIC TO LOWER CAMBRIAN

PCH: HYLAND: thin to thick bedded, brown to pale green shale, fine to coarse grained quartz-rich sandstone, grit, and quartz pebble conglomerate; minor argillaceous limestone; phyllite, quartzofeldspathic and micaceous psammite, gritty psammite and minor marble (Hyland Gp., Yusezyu)

PCH: HYLAND: grey weathering, dark grey to grey white, thin to thick bedded, very fine crystalline limestone, locally sandy; calc-silicate and marble; may locally include carbonate members(Hyland Gp., Algae Lake , limestone member of Yusezyu)

The Property is located on the western edge of the epicratonic Selwyn Basin, which is bound on the south-west by the Tintina Fault and on the north by the Dawson Thrust Fault (Gordey and Makepeace, 2001). The Selwyn Basin stratigraphy consists of late Proterozoic to Paleozoic marginal basinal and platformal clastic and pelitic sedimentary rocks derived from the North American Craton. Various aged volcanic and intrusive rocks are stratabound within, and intrude, the sedimentary package. During the Proterozoic and again in the late Devonian, the basin was subjected to rifting. This rifting was accompanied by volcanism and emplacement of thick sills of intrusive rocks.

By late Jurassic, the rocks of the Intermontane Belt of the Cordillera collided with the passive margin of the North America Shelf, causing compressive tectonics (Murphy, 1997). This resulted in crustal shortening, tight folding, and thrusting. Three regionally stacked thrust panels were formed which are separated by the Robert Service, Tombstone and Dawson thrust faults (from oldest to youngest) (Murphy, 1997). This thrusting has mainly affected the Intermontane and Omineca belts.

7.2 Local Geology

Meta-sedimentary rocks found on the Property are composed of Rabbitkettle Formation (Cambrian-Ordovician) calcareous phyllite overlain by Road River Group (Ordovician-Silurian) volcanic and off-shelf sedimentary rocks and Earn Group (Lower Devonian) shelf siliciclastic rocks. Throughout most of the Property, Cretaceous monzonite and quartz monzonite intrude Earn Group and Road River Group stratigraphy as a series of semi-conformable sills along a 15 km strike length. Cretaceous (91 Ma), Tombstone Suite biotite monzonite and syenite stock-like bodies occur locally in the south-central part of the Project. Sill emplacement is primarily localized within tectonized, graphitic argillite at the contact between the Earn and Road River Groups. This contact is also the locus of NNE-directed thrust faulting that has placed thin sequences of Silurian siltstone against Devonian siliciclastic units. The age of thrusting is probably related to the earliest Cretaceous movement on the Tombstone Thrust.

A property geology map and legend are shown in Figure 7–3 and Figure 7–4, respectively.

Figure 7–3: Property Geology Map

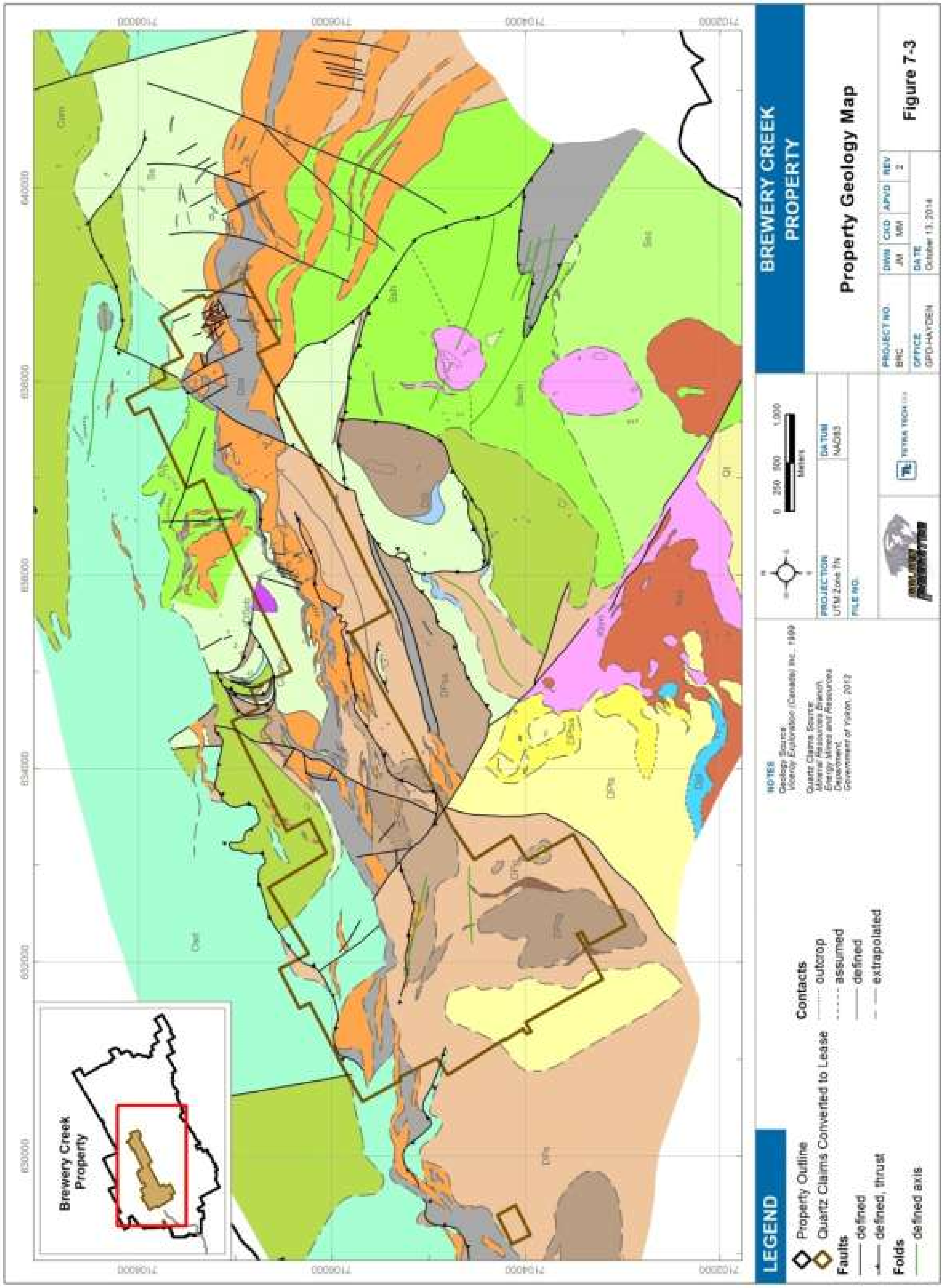
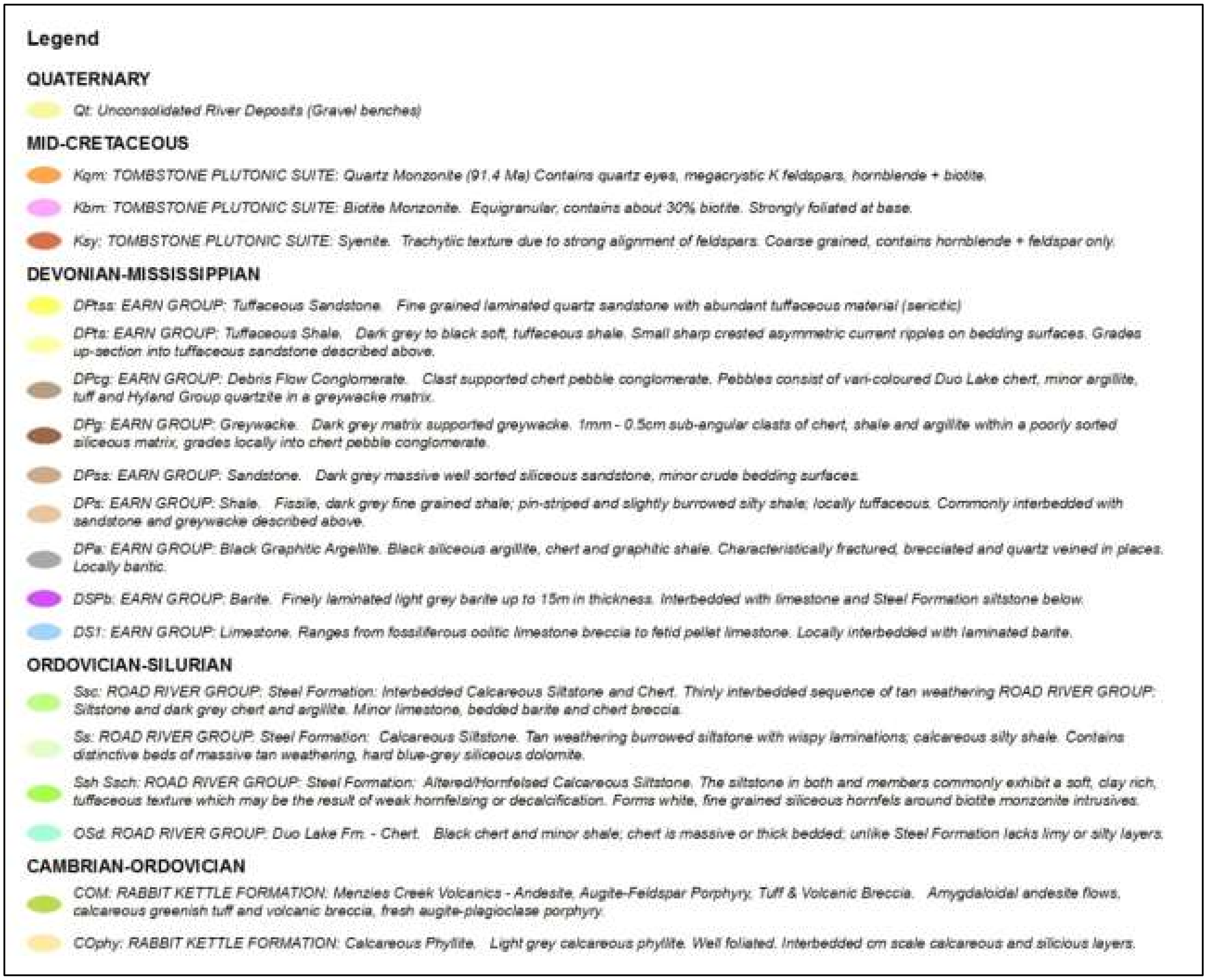


Figure 7–4: Property Geology Legend



7.2.1 Stratigraphy

Stratigraphy at Brewery Creek is comprised of sedimentary and volcanic rocks of the Selwyn Basin. Metasedimentary rocks include the Rabbitkettle Formation, Menzie Creek volcanic rocks, Road River group and Earn Group. Each of these lithologies is described in detail from oldest to youngest below.

Rabbitkettle Formation

The Rabbitkettle Formation consists of tightly folded calcareous phyllite and calcareous siltstone. The unit is thinly laminated, and is locally interbedded with chert and mudstone. This unit crops out in the Moosehead and North Slope zones in the north-central portion of the property. Though no age indicators have been identified in this formation on the property, it has been interpreted as old as Cambro-Ordovician (Gordey, 1981; Thompson et al., 1992) and as young as mid-Ordovician (Gordey and Anderson, 1993). The Rabbitkettle Formation was likely deposited in an area of tectonic stability, which received dominantly shallow water sediments deposited at low energy. Lindsey (2006) observed local cross-lamination and graded bedding suggesting the formation is upright.

Menzie Creek Volcanic Rocks

The Menzie Creek rocks consist of medium grained, chlorite and carbonate altered dolerite and basalt, hyaloclastite breccia and mafic volcaniclastic units. The Menzie Creek rocks lie unconformably over the Rabbitkettle Formation, and are overlain by the Steel Formation. Though no determination of age has been made for the Menzie Creek volcanism, Diment and Craig (1999) suggest a late Cambrian to early Ordovician age.

Road River Group

This stratigraphic unit is exposed throughout the Project area and is typically found along the northern portion of the property. It is made up of wispy laminated calcareous siltstone and massive chert conformably overlying the Rabbitkettle Formation. The Group is subdivided into the older Duo Lake Formation and the younger Steel Formation (Cecile, 1982).

Steel Formation

The Steel Formation is seen throughout the Property. This unit defines the top of the Road River group, and, may have acted as a focus of intrusion emplacement between overlying Earn Group rocks and underlying Road River rocks. The Steel Formation consists of wispy laminated siltstone with burrow marks, and interbeds of graphitic shale. Conodont assemblages in the Steel Formation have been identified as Silurian to early Devonian in age (Norford and Poulton, 1995). The formation also contains what appear to be turbidite sequences, or storm shelf debris flows, which may have been formed during a period of rifting.

Earn Group

The youngest package of sedimentary rocks on the property is the Earn Group, which unconformably overlies the Road River Group and represents platform, or shelf, marine sediments. The package is the primary host to the Brewery Creek Reserve Trend. This unit is composed of graphitic argillite, graphitic siltstone, argillite with lesser sandstone, greywacke, and chert-pebble conglomerate. Interbedded within the Earn Group, are black limestone and barite horizons. No age determinations have been made for the Earn group strata, but Campbell (1967) suggests that deposition of this unit extended from the Devonian through the early Carboniferous. It is likely that the Earn group was formed in an area of tectonic stability during periods of ocean transgression/regression sequences.

7.2.2 Intrusive Rocks

The majority of the gold mineralization at Brewery Creek is hosted within mid-Cretaceous, felsic intrusive rocks of the Tombstone Plutonic Suite. The intrusive rocks are exposed along an east-north-easterly striking structural zone over a distance of 15 km along strike and 0.5-2.0 km perpendicular to strike. Several compositional and textural phases have been mapped in drill core and drill cuttings. The older intrusive phases are emplaced parallel, or sub-parallel, to sedimentary bedding and along thrust faults often resulting in sill-like geometries, while the younger intrusive phases are present as dikes and small stocks distinctly discordant to the country rock. The sill complexes are the main host for gold mineralization, while the younger discordant intrusive rocks host lower grade gold mineralization. The thickness of the individual sills and the entire sill complex varies across the property from 100’s of metres in the southeast (Sleeman area) to 10’s of metres in the northwest (Pacific area). Some thicker sill complexes host volumetrically greater amounts of gold mineralization (Kokanee-Golden; Bohemian-Schooner areas).

The oldest intrusive rocks in the area are a series of monzonite and quartz monzonite sills. These rocks are fine to medium grained with textures ranging from equigranular to porphyritic. Phenocryst assemblages are comprised of variable amounts of biotite (5-30%), orthoclase (40-55%), plagioclase (30-40%) with minor quartz and hornblende. Biotite and orthoclase are commonly euhedral with phenocrysts ranging from 1-3 mm and 3-20 mm in diameter respectively. Large, zoned megacrysts of orthoclase with biotite inclusions are common in the southeastern portion of the property. Plagioclase is commonly subhedral with phenocrysts ranging from 3-10 mm in diameter. Xenoliths of black argillite are common in these rocks.

Sedimentary rocks on the margins of the sills are commonly strongly sheared suggesting that the sills followed older, low-angle structures. Locally, clasts of monzonite are incorporated into the shear zones defining a component of post-sill emplacement deformation. U/Pb isotopic dating of zircon from these monzonites yield an age of 91.4 Ma ± 0.2, similar to other Tombstone Suite intrusions in the region.

In the Sleeman area, younger monzonite dikes cut the older intrusions. The dikes are biotite bearing with no free quartz or hornblende and have a much finer grained texture. Where these dikes are altered the feldspars are converted to clay and biotite to white mica/clay.

South of the main sill complex are small stocks of biotite monzonite and syenite that intrude Road River Group and Earn Group sedimentary rocks. These intrusions are relatively coarse-grained with equigranular to porphyritic/pegmatitic textures. The stocks crosscut sedimentary bedding and local tremolite-epidote-diopside-garnet-skarn is developed marginal to the intrusive units. These intrusions host gold mineralization in the Classic and Lonestar areas.

7.2.3 Structural Geology

Paleozoic meta-sedimentary strata at Brewery Creek form a homoclinal sequence that strikes approximately 070° and dips moderately southeast. The sequence displays tectonic fabrics and geometries that indicate polyphase deformation including thrust faults that strike approximately parallel to stratigraphy and accompanying folds. Earlier workers describe multiple generations and orientations of folding (Lindsay, 2006; Diment and Simpson, 2009); work completed by Golden Predator has not verified these features. At least three orientations of high-angle faults formed subsequent to thrust faulting. Many of the fault sets described below, influence or control the distribution of mineralization.

Thrust Faults

Stratigraphic repetitions best define the positions of thrust faults at Brewery Creek. Many were mapped by earlier workers along the main area of mineralization (Diment and Simpson, 2009). The faults generally strike east-northeast (±070° AZ), dip moderately southeast, and commonly place siltstone of the Steele formation above variably graphitic and locally baritic argillite of the Earn group. Graphitic argillite typically occurs within and along the fault zones and defines the zone of displacement. The argillite typically displays well developed tectonic fabrics.

Regional work by Murphy (1997) shows that thrust faulting took place between late Jurassic and mid-Cretaceous time based on the age of the youngest stratigraphy cut by the thrust faults and a 142 ±6 ma date on muscovite in the Tombstone Strain Zone, a cross cutting structural feature. The Jurassic date is consistent with thrust faults mapped regionally in the Brooks Range (Plafker, 1994).

The Brewery Creek sill complex intrudes and lies concordant within proximity to the thrust faults but shows no direct evidence of intrusion syn-thrust faulting. Apparently, these sills are younger than the latest movement on the faults and appear to have utilized them as an intrusive plumbing system.

High-angle Faults

At least three orientations of high-angle faults occur at Brewery Creek, one set strikes northeast, another strikes northwest, and the other east-northeast; all are steeply dipping. The northeast and northwest striking sets show a strong component of strike displacement and commonly displace mineralization. The east-northeast striking structures show primarily normal displacement.

Northwesterly striking structures generally have a strike azimuth of approximately 330° and are near vertically dipping. Relations visible in the Kokanee open pit, show dextral displacement of mineralization. They commonly have local displacements of 3-10 metres, however, field relations suggest overall displacement up to a few hundred metres. Lindsay (2006) suggests dextral movement along the 300° azimuth striking Classic Fault could have produced 1.5 km of dextral displacement.

North-easterly striking structures have azimuths of 020° to 030° and are generally near vertical dip. Fault fabrics indicate that the primary direction of displacement is strike-slip. Where confirmed by local outcrop relations, they show sinistral displacement. The magnitude of total displacement is difficult to interpret since they generally strike semi-parallel to the lithologies and mineralized zones.

East-Northeast striking faults occur throughout the district. They generally have an azimuth of 070° and dip steeply to the northwest. Outcrop relationships in the Kokanee open pit show that they are normal faults that displace rocks down to the north-northwest. Displacement is generally small; where observed in outcrop, less than 10m. Closely spaced joint sets commonly parallel these faults.

Tectonic fabrics within fault zones exposed in outcrop demonstrate that the northeast and northwest structures were co-active, and their orientations are consistent with a conjugate set. These faults cut the sill complex representing the most recent movement subsequent to sill intrusion at approximately 90 to 92 ma. Minor and small-scale quartz-sulfide veinlets and stockworks with 330° azimuths were observed in the hangingwall of a northeast-striking fault in the Golden deposit, suggesting that the 330° azimuth orientation was active during mineralization. No major mineralized zones, except Classic, follow the northwest orientation, indicating that, though active, it was not strongly dilatant during mineralization. Several large deposits and mineralized zones, including parts of the Kokanee and Golden deposits, follow mapped northeast-striking faults, indicating that the northeast orientation was active and strongly dilatant during mineralization. The northwest-striking faults show the greatest amount of post-ore displacement.

7.3 Mineralization

Historic production on the property occurred along the historical Brewery Creek Reserve Trend (BCRT). The Brewery Creek Property consists of numerous in-situ deposits, mineralized zones and past producing deposits both along this trend as well as within peripheral mineralized areas. Past producing areas within the BCRT include the Pacific, Blue, Canadian, Upper Fosters, Kokanee, Golden and Lucky deposits. Additional to these, unexploited mineral resources have been defined for the Big Rock West, Big Rock East, Lower Fosters, Bohemian and Schooner deposits along the BCRT; the North Slope deposit north of the BCRT; Sleemans deposit east of the BCRT, and the Classic and Lonestar deposits south of the BCRT. The Moosehead (some minor historical production) and Ice Fog zones also lie north of the BCRT.

7.3.1 Areas Included in PEA

Big Rock

The West and East Big Rock deposits are the furthest westerly known occurrence in the district and are located approximately 1.2 km from the current heap leach pad. They were discovered in the early 1990s by Viceroy Gold by soil sampling and trenching. The two zones were first drilled in 1991; most of the historical drilling was carried out between 1994 and 1998, with some recent drilling completed by Golden Predator. The deposits are defined by 213 reverse-circulation rotary holes, and 69 core holes, totaling 22,288 metres of drilling. The West Big Rock deposit is ~650 metres in length, ~30 metres wide, and ~220 metres down dip. The East Big Rock deposit is ~640 metres in length, ~30 metres wide and ~180 metres down dip.

Mineralization occurs primarily in limonite-altered quartz monzonite sills and subordinately in adjacent siliciclastic sedimentary strata. Big Rock sills strike 070° azimuth and dip between 40 and 45 degrees southeast and have a drill-defined strike length of approximately 1.5 km. The eastern part of the sill complex and deposit are truncated by the Classic fault, or a splay. Lindsay (2006) suggests that Big Rock mineralization is a westerly continuation of the BCRT that is displaced approximately 1.5 km to the northwest by the Classic Fault. An alternate interpretation is that the Big Rock resource is a westerly continuation of mineralization along the Ice Fog and North Slope mineralized zones. No other faults were mapped or modeled in the Big Rock resource area.

The reverse-circulation drilling chip logs show that gold mineralization occurs primarily in clay-altered quartz monzonite. Much of the zone is oxidized, and the location of oxidation from surface down suggests that it resulted from supergene processes. The distribution of elevated gold values with respect to sill-form intrusions suggest that lithology, and perhaps rock rheology was a primary control on mineralization.

Canadian and Fosters

The Fosters mineralized area includes only the un-mined Lower Fosters deposit, which lies approximately 3.5 km from the current heap leach pad. The Upper Fosters and Canadian deposits have been mined historically and are not considered as part of this PEA. The area is defined by 392 reverse-circulation drillholes and 40 core holes, totaling 19,550 metres of drilling. Numerous blastholes were drilled within the historical pits for which location and analytical data exists. The Lower Fosters deposit (considered as part of PEA) is ~550 metres in length, ~30 metres wide, and ~260 metres down dip.

A large sill complex extends throughout the Fosters-Canadian area and hosts most of the known mineralization. It has a strike length of at least 1.2 km and a down-dip extent of at least 500 metres. It strikes 070° azimuth and dips approximately 20° southeast. The sill complex contains large interleaves of sedimentary strata and splits into a complex array of individual sills along strike and dip.

Several faults traverse the area. Modeling shows that a 330° azimuth fault offsets the western extension of the Canadian deposit, and a 020° azimuth Fault separates the Canadian deposit from the Lower Foster's deposit. Logged gouge zones in several holes along the northernmost known extent of the sill complex indicates that a major 070° azimuth fault may offset the down dip continuation below the Lower Fosters resource.

Logs of reverse circulation drillholes indicate that mineralization associates with clay alteration, presumably from the destruction of K-feldspar minerals. According to Diment and Simpson (2009), mineralization is associated with pervasive phyllic and locally intense argillic alteration. The feldspars alter to an assemblage of sericite, illite and kaolinite. Fine pyrite and arsenopyrite occur in association with secondary quartz. Gold occurs primarily in the limonite-altered quart monzonite and subordinately in sedimentary strata that lie adjacent to the intrusions.

Bohemian-Schooner

The Bohemian-Schooner deposit and surrounding mineralized area was originally discovered by soil sampling, trenching and drilling in the 1990s by Viceroy. The area remains unmined and is defined by 129 reverse-circulation drillholes and 122 core drillholes, totaling 23,385 metres. The Bohemian deposit is ~520 metres in length, ~50 metres wide, and ~160 metres down dip. The Schooner deposit is ~450 metres in length, ~50 metres wide, and ~160 metres down dip. A linear distance of approximately 7 km separates these zones from the old heap leach pad.

A sill complex at Bohemian-Schooner hosts the majority of mineralization. It intrudes a section of siltstones of the Steele Formation and interleaved, structurally dismembered carbonaceous argillite of unknown affinity. The composite strike length of the sill complex is over 1 km oriented east-west, dipping 5° to 10° to the south. A prominent high-angle east-west striking structural zone traverses the entire length of the area. Sills occur on both sides of the structure and are displaced down to the north across it. The sills are thickest along the structure, indicating that it may have localized the intrusions. Higher grade parts of the resource also align along this structure.

A large fault with a 330° strike azimuth lies between the Bohemian-Schooner resource area and the formerly mined, Lucky deposit to the west. Sporadic mineralization and isolated drill intercepts in the intervening area between these two areas indicates that they may have been contiguous prior to faulting. If so, the fault would have a total displacement of over 250 metres. Alternatively, if the fault displaced farther, the Bohemian-Schooner resource could have aligned with the eastern extension of the North Slope – Ice Fog trend. Much of the section at Bohemian-Schooner consists of siltstone of the Steel Formation, also suggesting a possible affinity with the North Slope – Ice Fog trend.

Gold mineralization at Bohemian-Schooner occurs primarily in clay-altered quartz monzonite sills and subordinately in adjacent siltstone. It occurs most commonly in association with strong argillic altered and locally silicified quartz monzonite. Sheeted and stockwork mm- to cm-scale quartz-pyrite-arsenopyrite veins, commonly forming conjugate patterns in detail, cut the altered intrusion and occur in association with higher grade zones.

Kokanee

The Kokanee deposit was mined by Viceroy from four pits; all pits were partially backfilled. The southern two pits remain mostly open while the northern two are almost entirely backfilled and reclaimed. The deposit is centrally located along the BCRT and formed in the thickest and most extensive part of the Cretaceous quartz monzonite sill complex. The deposit is defined by 31 core holes and 506 RC holes, totaling 29,654 metres. The deposit is ~1100 metres in length, ~40 metres wide, and ~190 metres down dip.

Mineralized material at Kokanee occurs primarily in the quartz monzonite sill complex and subordinately in siltstone and argillite. Observations of mineralized material exposed in pit walls shows millimeter-scale veinlets with iron-oxide ± quartz fillings. The mineralized quartz monzonite typically contains several percent of evenly disseminated oxidized pyrite.

Drill logs indicate alteration of the K-feldspar component of quartz monzonite to white clay. Locally developed auriferous sheeted quartz veins were noted in pit highwalls. Pervasive silicification occurs locally, but is not common.

Golden

The Golden deposit lies immediately east of Kokanee and may be a faulted offset of Kokanee. It was mined by Viceroy from 4 pits; three were backfilled and reclaimed, the lowest and farthest south pit was not backfilled and remains in its fully mined state. The deposit is defined by 19 core holes and 363 RC holes, totaling 21,251 metres. The deposit is ~950 metres in length, ~30 metres wide, and ~150 metres down dip.

Golden, like Kokanee, is hosted by the thickest and most extensive part of the Cretaceous quartz monzonite sill complex. It is a nearly identical system structurally, and the styles of alteration identical. Both of these resource areas show a bi-directionality to the strike direction of the highest grade ore, one northeast and the other northwest trending, forming a conjugate pattern.

The K-feldspar component of quartz monzonite, both phenocryst and groundmass are altered to white clay. Locally developed auriferous sheeted quartz veins and seems filled with oxidized Fe were noted in pit highwalls. Pervasive silicification occurs locally, but is not common. The most pervasively developed alteration occurs along faults with orientations similar to the distribution of higher grade material, suggesting that these structures were hydrothermal fluid conduits.

Lucky

The Lucky pit was mined by Viceroy, partially backfilled and reclaimed. The deposit occupies the northeastern-most segment of the BCRT. It is situated immediately west of the Bohemian-Schooner deposits and northeast of the Golden deposit. The Lucky deposit is defined by 169 RC drillholes and 3 diamond drillholes, totaling 11,240 m. The deposit is ~550 metres in length, ~50 metres wide, and ~360 metres down dip.

Altered Cretaceous quartz-monzonite that intrudes lower Earn Group sediments host mineralized material at Lucky, similar to that at Bohemian-Schooner. Dominant mineralized trends typically strike 035° or 060° and dip moderately (-25 to -45) to the southeast. Mineralized material in the hanging wall is abruptly terminated to the northwest by Steel-formation sediments at the footwall contact of a major 040° trending fault.

7.3.2 Areas Not Included in PEA

Pacific

The Pacific deposit was mined by Viceroy; the pit was not backfilled, and remains in its fully mined state. Pacific lies along the Reserve Trend, immediately east of the Classic Fault. The deposit is defined by 17 core holes and 80 RC holes, totaling 6,966 metres. The deposit is ~500 metres in length, ~50 metres wide, and ~300 metres down dip.

Pacific is the only deposit in the district that is hosted primarily by lower Paleozoic siltstone. Mineralization is generally tabular and follows a combination of shallow south dipping bedding and high-angle BCRT-parallel faults. Higher grade parts of the deposit are steeper along these faults. The deposit has been segmented by several post-mineralization northwest-trending dextral faults.

Observations of mineralized material exposed in pit walls shows millimetre-scale veinlets with iron-oxide ±quartz fillings. One occurrence was noted of a pervasively silicified breccia at the intersection of a northeast-trending and a northwest-trending set of faults. The breccia contains angular fragments of silicified siltstone in a quartz matrix.

Blue

The Blue deposit was mined by Viceroy, and the pit was partially backfilled and reclaimed. Blue lies directly east of the Pacific deposit along the BCRT. A fault separates the two deposits; one possible restoration of displacement suggests that the two deposits may have been a single mineralizing system. The deposit is defined by 26 core holes and 113 RC holes, totaling 8,149 metres. The deposit is ~560 metres in length, ~45 metres wide, and ~200 metres down dip.

Blue is hosted primarily by Cretaceous quartz monzonite and subordinately by lower Paleozoic siltstone. Mineralization is generally tabular and follows the strike and dip of the sill complex. Unlike Pacific, the primary strike of the deposit lies along a series of northeast-trending faults. A strong discontinuity in stratigraphy, sill development, and mineralization occurs at the eastern end of the deposit. An area of poorly defined mineralization occurs immediately southeast of the deposit, suggesting a possible post-mineralization offset of the deposit along a northwest trending fault.

Drill logs indicate that alteration of the quartz monzonite includes strong white clay development after K-feldspar phenocrysts and groundmass, and locally developed auriferous sheeted quartz veins. Pervasive silicification is noted locally, but is not common.

Moosehead

Moosehead was the last deposit mined by Viceroy. The pit was not backfilled and remains open. Material from minor slope failures fills the bottom of the pit and benches. The Moosehead deposit is defined by 163 RC drillhole drillholes and 14 diamond drillhole drillholes, totaling 10,530 m. Golden Predator Canada Corp. drilled 10 core holes totaling 1,180 m in 2012 in an effort to better understand the metallurgy of the zone. No mineral resources are currently recognized at Moosehead.

An altered northeast-striking, southeast dipping, quartz monzonite sill hosts gold mineralization. Mineralized material occurs as veinlets and disseminations within and along the margins of steep south-dipping fractures. The deposit is structurally truncated to the west and down dip; a fault that juxtaposes graphitic siliciclastic sedimentary strata with the quartz monzonite sill truncates mineralization at depth, and a high-angle fault truncates the western extension.

North Slope

The North Slope deposit lies approximately 1 km north of the deposits of the BCRT, and approximately 4 km from the heap leach pad. The zone lies conformably within a lower stratigraphic section than the BCRT. It was initially discovered by soil sampling, trenching and drilling carried out by mine personnel during the 1990's by Viceroy Minerals. Golden Predator renewed exploration efforts by drilling core holes in 2009, and continued core and RC drilling in 2011. The deposit is defined by 108 reverse-circulation rotary holes, and 32 core holes, totaling 24,221 metres of drilling.

The mineralized zone occurs in clay-altered quartz monzonite and siltstone of the Steele Formation, lower in the stratigraphic section than most of the mineralization along the BCRT. The current drilled extent of the structure and sill complex at North Slope is 640 metres along strike and approximately 500 metres down dip, with mineralization intersected at up to 700 metres down dip. It strikes 070° azimuth and dips approximately 40° southeasterly. The mineralized sills and structural zone remain unconfined along both strike directions. To the northeast, the zone strikes toward the Ice Fog zone.

Geologic observations in core suggest that mineralization occurs within and along a continuous and through-going breccia zone that strikes and dips parallel to the structures in the BCRT. This breccia zone may define a thrust fault that was later intruded by the sills.

Gold mineralization is spatially associated with carbonate/clay + quartz alteration in both siltstone and intrusive lithologies. Multiple stages of arsenic-poor pyrite and marcasite are present in the mineralized zones and arsenopyrite is present as discreet crystals on the surface of the earlier pyrite. Visible gold has not been observed, but may be associated with the later arsenopyrite mineralization.

Sleeman

The Sleeman deposit is located to the east of the BCRT and may possibly demarcate the easternmost extent of the trend. It was discovered by mapping, soil sampling and trenching, and was first drilled in 1992. The zone is currently defined by 7 reverse-circulation drillholes and 58 core drillholes, totaling 11,374 metres. A linear distance of approximately 9 km separates the zone from the heap leach pad. The deposit is ~500 metres in length, ~25 metres wide, and ~220 metres down dip.

Mineralization at Sleeman is associated with an altered tabular-shaped quartz monzonite intrusion that cuts siltstone of the Steel formation and graphitic argillite of unknown affinity. The intrusion strikes 120° azimuth and dips 65° southwest. It has a known strike length of 500 metres and is open in both strike directions and at depth. A secondary trend of mineralization oriented approximately 060° azimuth and dips approximately 45° to the southeast is noted in the western hanging wall to the main tabular body. A poorly constrained fault may displace the southeast portion of the sill down to the southeast.

Alteration at Sleeman includes locally intense clay development after feldspars and texture destructive silicification. All mineralization is associated with the altered and veined areas. Hairline to millimetre-scale quartz-pyrite stockworks and planar 2-10 millimetre-scale quartz-pyrite veins with illite selvages occur within the alteration envelope. The planar quartz veins are paragenetically younger than the stockworks. The style of veining and alteration at Sleeman is similar to the other deposits found within the BCRT with the exception of the presence of elevated base metal concentrations, particularly lead and zinc.

Classic

The Classic deposit is located approximately 3 km south of the main BCRT, 7 km west of the Sleeman deposit and 4 km south of the old heap leach pad. Discovered originally in 1991 (Hemlo Gold Mines Inc.-Loki Gold Corporation) through a southern grid expansion, the Classic Zone was then being classified as an isolated, arsenic gold anomaly. To date, the Classic deposit remains a poorly understood with current interpretations based on the underlying pluton and structural faulting. It is currently defined by 52 reverse-circulation drillholes and 17 core holes, totalling 13,478 metres. The currently identified mineralization lies entirely on the southwest side of the Classic fault. The deposit is ~1400 metres in length, ~30 metres wide, and ~240 metres down dip.

Predominant rock units hosting mineralization contain variable percentages of syenite (alkali) and biotite monzonite (increasing plagioclase). Mineralization is found to exist within centimetre-scale sheeted quartz veinlets. Structurally, the Classic zone is open at depth and in both directions along strike. Cutting across the eastern portion is the northwest trending and steeply south west dipping Classic fault which is mapped to be post intrusion and post mineralization. A similar intrusive complex which displays altered mineralization akin to the Classic is mapped within the footwall of the Classic fault with a dextral offset of 1.5 km (Lindsay, 2006) to the southeast.

Lone Star

The Lone Star mineralized area lies along the northeast side of the Classic fault, southeast of and adjacent to the Classic Zone. Surface mineralization was first recognized by soil sampling in the 1990's but the area remained untested until 2012. Drilling in 2012 consists of 17 core holes and 12 RC holes, totaling 6,147 metres. The deposit is ~1100 metres in length, ~20 metres wide, and ~220 metres down dip.

The same alkalic suite of intrusions that host Classic also host Lone Star. The suite intruded along a zone with an azimuth of 290°, centered on and sub parallel to the post-mineralization Classic fault. The suite contains syenite, biotite monzonite, monzodiorite, diorite, and gabbro; syenite is the most abundant. The more mafic compositions intrude the syenite and the most mafic lithologies were last to intrude. The biotite monzonite intrusions commonly form very well developed, course-grained skarn halos where adjacent to limestone.

Alteration includes development of a propylitic mineral assemblage of chlorite, calcite and pyrite, and local development of sheeted quartz-carbonate-pyrite-arsenopyrite ±chalcopyrite veins. Three styles of mineralization occur at Lone Star; elevated Au associated with skarns, disseminations in syenite, and auriferous sheeted quartz veins. The geometry of the system is poorly understood; it remains open in both strike directions and at depth.

7.4 Local Surficial Geology, Terrain and Permafrost

A reconnaissance level site assessment of surficial geology, terrain hazards, surficial soils and areas of potential permafrost extent was undertaken by Tetra Tech EBA in 2012.

The Brewery Creek Property is situated at the southern extent of the Mackenzie Mountains Ecoregion at the edge of the Tintina Trench and the Yukon Plateau-North Ecoregion to the south. The region was not subjected to continental glaciation in the last (Wisconsin) glaciation, resulting in terrain characterized by narrow, v-shaped stream valleys that often follow the fracture zones of fault lineaments, and deep weathering of bedrock.

The Ogilvie Mountains act as an orographic barrier in the region to air masses moving off the Gulf of Alaska inland, generating a wet belt, particularly along the southern slopes. These Mountains also stop shallow layers of cold arctic air from reaching the southern and central Yukon. Mean annual temperatures in the Ecoregion are near –6°C with temperatures of -25°C in January and 8°C in July. The study area lies within the zone of discontinuous permafrost with regular occurrence likely above an elevation of 1,500 masl and underneath most valley bottoms (Brown 1978; Yukon Ecosystems Working Group, 2004).

The terrain configuration is low-relief rolling to undulating upland, deeply incised by a network of moderately steep-sided, V-shaped stream valleys. Natural rock outcrops are uncommon; however, large areas of bedrock are exposed in the open pits of mined areas and on some road cut slopes. The study area is at or near the local height-of-land and stream catchment areas are small (Figure 7-5).

Colluvial deposits cover almost 100% of the natural terrain. The long period of exposure of surfaces to weathering, frost-shattering and soil creep has resulted in well-developed colluvial veneers on most surfaces overlying weathered bedrock (regolith). Soil texture of colluvium is typically granular. In areas underlain by quartz monzonite bedrock, colluvium was typically gravelly sand. Silt typically forms an increased proportion of the soil content in areas underlain by shale.

In many areas the colluvial veneer (up to 1 m thick) is overlain by a thin veneer (typically less than 0.3m thick) of organic soil, humus and the dense root layer of ground vegetation and peat. Some thickening of organic soil cover is expected on the lower slopes and floors of stream valleys.

Bedrock observed during field checking was mostly shale and quartz monzonite. The study area is within a region that was unglaciated during the last (Late Wisconsin) glaciation (Duk-Rodkin, 1995). Overburden in these unglaciated regions typically consists of a veneer (up to 1 m thick) to blanket (1 m to 3 m thick) of deeply weathered bedrock underlying colluvium. Near-surface shale was very fissile and soft. Shallow quartz-monzonite was friable and near the surface formed regolith (loose sand and gravel fragments of remnant rock particles).

The study area was generally moderately well drained, correlating with the typical coarse-textured colluvium and underlying regolith. Moderately drained areas were observed at some lower slopes and floors of stream valleys, often associated with those areas more likely to be underlain by permafrost.

Figure 7–5: Photo from helicopter of the Brewery Creek Property, looking east with reclaimed Canadian and Kokanee deposits along left side and Laura Creek along valley bottom (Golden Predator, 2012)

7.4.1 Terrain Hazards And Permafrost

Geomorphological processes include slow soil creep on the middle to lower slopes of some stream valleys and minor sloughing along some eroded stream banks. There were no indications of active rapid mass movement processes observed during the field visit. Some minor sloughing on fill slopes of existing waste piles appears to have occurred in the past, but these do not appear to be active. Some sloughing and sliding of a minor volume of organic-rich overburden at a new exploration road and drill site in the Bohemian-Schooner area was reported.

The majority of the area is judged to be free of permafrost; however, subsurface data to confirm permafrost is limited. Permafrost in the study area is discontinuous and is probable on most lower slopes and floors of the moderately steep (50% to 70% gradient), V-shaped stream valleys. There is a moderate to high probability of permafrost on north-facing, mid-elevation and some upper-elevation stream valley side slopes. Vegetation indicators of permafrost are thick moss ground cover and forest cover dominated by Black Spruce, often showing signs of stress (leaning or toppling) due to the limited rooting depth in a shallow active layer.

Field verification in hand test pits at field stations established during ground-truthing, and in hydrogeological investigation boreholes, confirmed the presence of discontinuous permafrost on lower elevation slopes and floors of stream valleys and on mid-elevation to upper-elevation north-facing slopes.

Permafrost degradation in fine-textured, ice-rich soils can result in unstable slopes. However, soil profiles observed on road cuts and in hand test pits indicate that surficial material below the thin veneer of organic soil is generally coarse-textured and thus is expected to be ice-poor and relatively thaw-stable. Waste rock dumps of coarse sand, gravel and boulders placed during historic mining appear relatively stable; however, most were placed on south-facing slopes where the likelihood of permafrost is low.

8.0 DEPOSIT TYPES

8.1 Mineral Deposit

The Brewery Creek deposits exhibit characteristics of both intrusion-related and epithermal type deposits. It is generally considered to be an alkalic intrusion-associated, gold deposit as most of the mineralization is concentrated within or proximal to the monzonites. Geological, geochemical, petrographic and fluid inclusion data indicate that original sill emplacement, first stage alteration and associated mineralization occurred at a relatively low temperature and high level within the crust. However, the presence of wispy-textured quartz veinlets, related to later shear zone deformation, indicates deposition at moderate to deep levels (Dunne, 1995), a common characteristic of epithermal type deposits (Poulsen,1996).

The following factors support an epithermal, environment of mineralization for the deposits along the reserve trend: 1) a strong gold, arsenic, antimony, mercury association within veins and breccias, 2) very low base metal concentrations and a relatively high gold : silver ratios of 3:1, 3) the absence of contact metamorphism in sediments around sill contacts along the reserve trend, 4) euhedral, coarse-grained quartz with primary growth zones, 5) open space textures such as comb and cockade textured quartz and chalcedony, and 6) the presence of trace amounts of CO2, low salinities (<7% NaCl) and low homogenization temperatures (< 300 °C) within fluid inclusions.

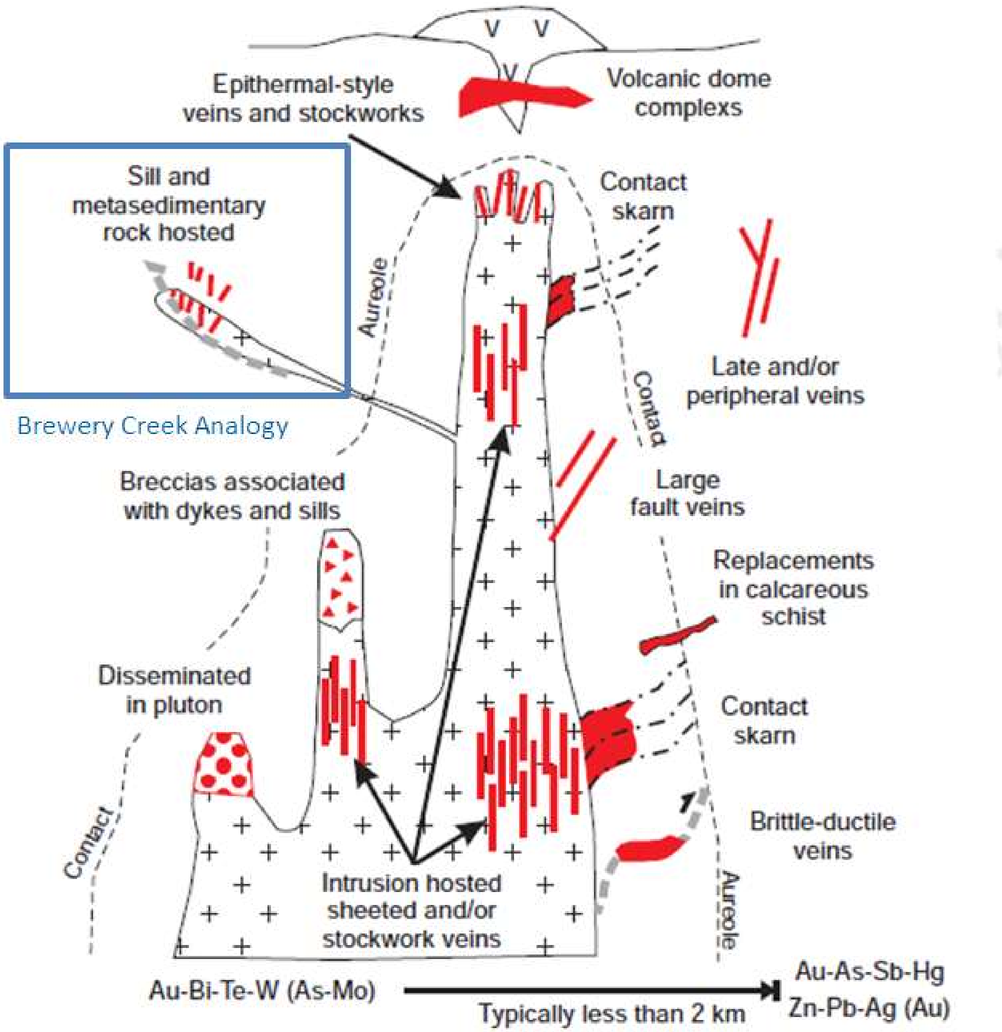
The mineralization delineated to date consists of fracture-controlled quartz stockwork in siliciclastic and intrusive rocks; however, the presence of local decalcification and silica replacement in the calcareous Steel Formation suggests that an epithermal type model may be appropriate along the Reserve Trend. Classic and Lone Star, and perhaps Sleeman, show alteration styles and patterns more consistent with a porphyry style of mineralization. The close association of mineralization with specific intrusive phases and strong skarn development suggest that these resources may have formed deeper and at a higher temperature than the reserve-trend occurrences.

8.2 Geological Model Applied

The Brewery Creek deposits resulted from mineralization that followed emplacement of an alkalic intrusive suite. Textures and styles of alteration show that they formed in an epithermal and perhaps sub epithermal environment. Gold and associated arsenic and antimony mineralization are hosted by both intrusive and sedimentary lithologies as depicted in Figure 8–1.

This model is very similar to gold deposits described for the ACMA-Lewis Deposit at Donlin Creek, Alaska, where significant resources of sulfide related gold mineralization are currently being evaluated by NovaGold Resources, Inc. The mineralization style, alteration characteristics, age and scale of the mineralized zones seen at Brewery Creek are similar to those described by Hanson et al (2009). Brewery Creek is attributed to the sill and meta-sedimentary rock-hosted style as shown below in Figure 8–1 (extracted and modified from Lindsay, 2006).

Figure 8–1: Geological Model Schematic (Extracted and Modified from Lindsay, 2006)



9.0 EXPLORATION

9.1 Recent Exploration Work

Exploration conducted by Golden Predator since 2009 includes geophysical surveys, soil sampling surveys, and extensive drilling campaigns. These surveys were undertaken to extend known mineralized zones, reveal new mineralized zones, and provide information on parts of the property which had not been tested.

In 2011, Golden Predator contracted Fox Exploration to conduct soil sampling of the Classic zone and on new extensional claims on the property. During 2011, Aurora Geosciences of Whitehorse, YT was contracted to conduct an IP survey over the Sleeman zone at the eastern portion of the property.

Exploration on site conducted by Golden Predator in 2012 included geophysical surveys and an extensive drilling campaign. As the exploration model changed from prior ownership, these additional surveys were undertaken to extend known mineralized zones, test new exploration targets, and provide information on parts of the property which have had no work done. In 2012 Precision GeoSurveys Inc. of Vancouver, BC was contracted to fly an airborne magnetic survey in an effort to better define the magnetic signatures of known intrusive host rocks in the Classic and Lone Star areas. In addition to the airborne survey, a ground magnetic survey was performed over the Classic and Lone Star areas.

In addition to the magnetic surveys, drilling was conducted throughout the West Big Rock, East Big Rock, Moosehead, Lower Fosters, Bohemian, Schooner, Classic and Lone Star areas. The 2012 drilling program comprised a number of tasks, including exploration drilling, resource addition/infill drilling, metallurgical characterization, column stack leachability testing, geotechnical drilling and water monitoring.

9.1.1 Airborne Magnetic Surveys, 2011-2012

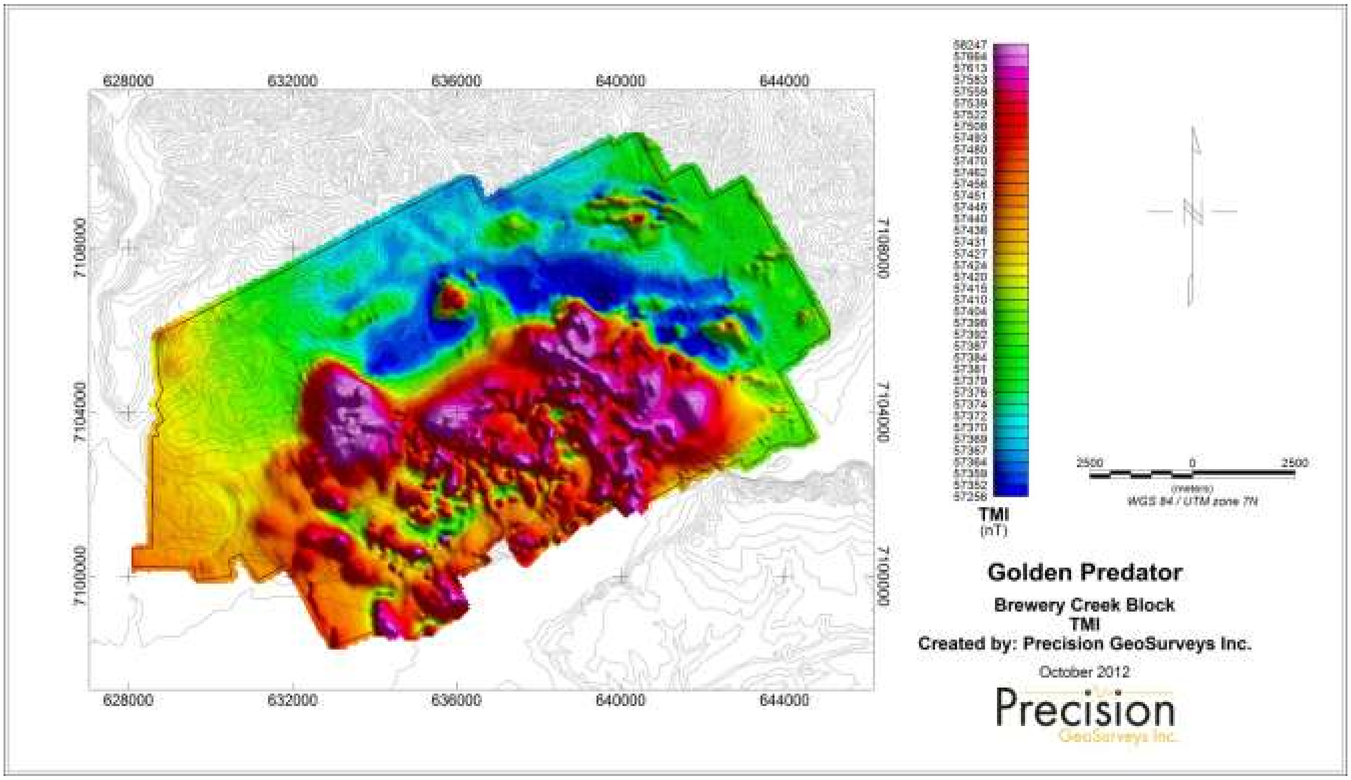
In 2011, Precision GeoSurveys Inc. of Vancouver, BC was contracted to fly an airborne magnetic survey. This was done in order to better define the magnetic signatures in known areas of mineralization along the BCRT over the Sleeman, Bohemian and Schooner portion of the property and to investigate these same signatures in unexplored areas.

In 2012, an additional airborne magnetic survey was flown in an effort to complete the 2011 survey over the full Brewery Creek claim block. The 2012 survey tied in with this previous survey. A total of 1064 km of flight lines were flown. Survey lines were located at 100m spacing's oriented east west, and tie lines were flown at 1 km spacing's oriented north south. Nominal survey height was flown at 35m above ground level and was flown with a Eurocopter AS350 helicopter. PEIComp, was used to create a model from the compensation flight data, and was then applied to the raw magnetic data to remove the noise.

The finalized data sets delineated a number of high and low magnetic signatures of interest on the property. The BCRT signature is characterized by a linear east/west magnetic low. This signature is likely the result of hydrothermal fluid flow through the intrusive body which results in the destruction of ferro-magnesium minerals in the intrusive body. The southern portion of the property shows a number of vast magnetic highs. Magnetic highs which have been tested by Golden Predator display strong linkages to the mineralization at the Classic and Lonestar zones (Figure 9–1). Both these zones host gold mineralization within vast syenite/alkali feldspar syenite/biotite monzonite stock and sill complexes. It is believed that these zones have not undergone the same intensity of ferro-magnesium destructive fluid flow as the reserve trend, and thus have retained their magnetic properties relating to presence of magnetite or pyrrhotite.

Numerous potential exploration targets were also delineated as a result of the survey.

Figure 9–1: Airborne Magnetics, Total Magnetic Intensity over Brewery (Precision, 2012)



9.1.2 Ground Magnetic Survey, 2012

In addition to the airborne magnetic survey, Golden Predator staff undertook a ground magnetic survey over the south eastern portion of the Classic zone and portions of the Lone Star zone in 2012. Precision GeoSurvey Inc. provided the ground magnetometer for use, as well as set up two magnetic base stations to ensure that diurnal activity was recorded during the survey. Two GEM GSM 19T base stations were set up well within the survey area. Base station readings were reviewed at the end of the day to ensure that no data were collected during periods with high diurnal activity (greater than 5 nT per minute). The base station was installed at a magnetically noise-free area, away from metallic items. Data was reviewed daily to ensure accuracy, and then sent to Precision GeoSurvey for post processing.

Due to inclement weather, the full proposed ground survey was cut short. However, the data provided helped define the magnetic signature of the mapped Classic fault, and re-affirmed the magnetic signature of the Classic and Lone Star intrusive rocks.

9.1.3 Induced Polarization Survey, 2011

An induced polarization (IP) survey was conducted by Aurora Geoscience in 2011. The survey covered a line distance of 19.8 km representing an area of approximately 4.3 km² over the Sleeman Zone. Lines were cut and picketed using handheld GPS units, which were also used to mark electrode and current injection points. Modified pole dipole arrangement of the electrodes was used for this survey with dipole spacing at 50m on all lines. The survey started with 50m – 10 conductor cables until the temperature dropped below -10 degrees Celsius. From there, the survey was done with a 10 channel – 500m wire bundle until the terrain became too steep and the snow too deep. The survey was then finished with 50 m – 6 conductor cables with a 4 channel – 200 m wire bundle.

The results of the IP survey over the Sleeman zone delineated chargeability high over the main zone of the Sleeman area, and northwest of the current drilling. There is also a resistivity low, which at near surface is quite narrow and linear.

9.1.4 Soil Sampling Survey

A 2011 soil sampling survey was conducted at the southern portion of the property, and the eastern claim extension including the Sleeman zone. The sample program was an in-fill program to obtain closer spaced data points in between earlier soil sampling events. Samples were collected at 50 metres spacing along soil lines 100 metres apart; lines over the Sleeman main zone were 50 metres apart. The southern soil survey covered approximately 9 km², and the Sleeman/claim extension covered approximately 7.4 km². Procedures were in place for collecting in areas of great talus cover, and duplicate samples were taken to ensure sample quality. A total of 4,305 samples were collected over the area including duplicates.

The soil sampling survey was considered successful in both areas by Golden Predator. The previous lack of samples due to permafrost at the Classic zone was overcome due to a recent forest fire, which exposed soils and reduced much of the near surface active permafrost.

The combined results of all the soil sampling programs refined the Lone Star area anomaly, refined scattered anomalies between Lone Star and Sleeman and highlighted some low level anomalies east of the BCRT.

10.0 DRILLING

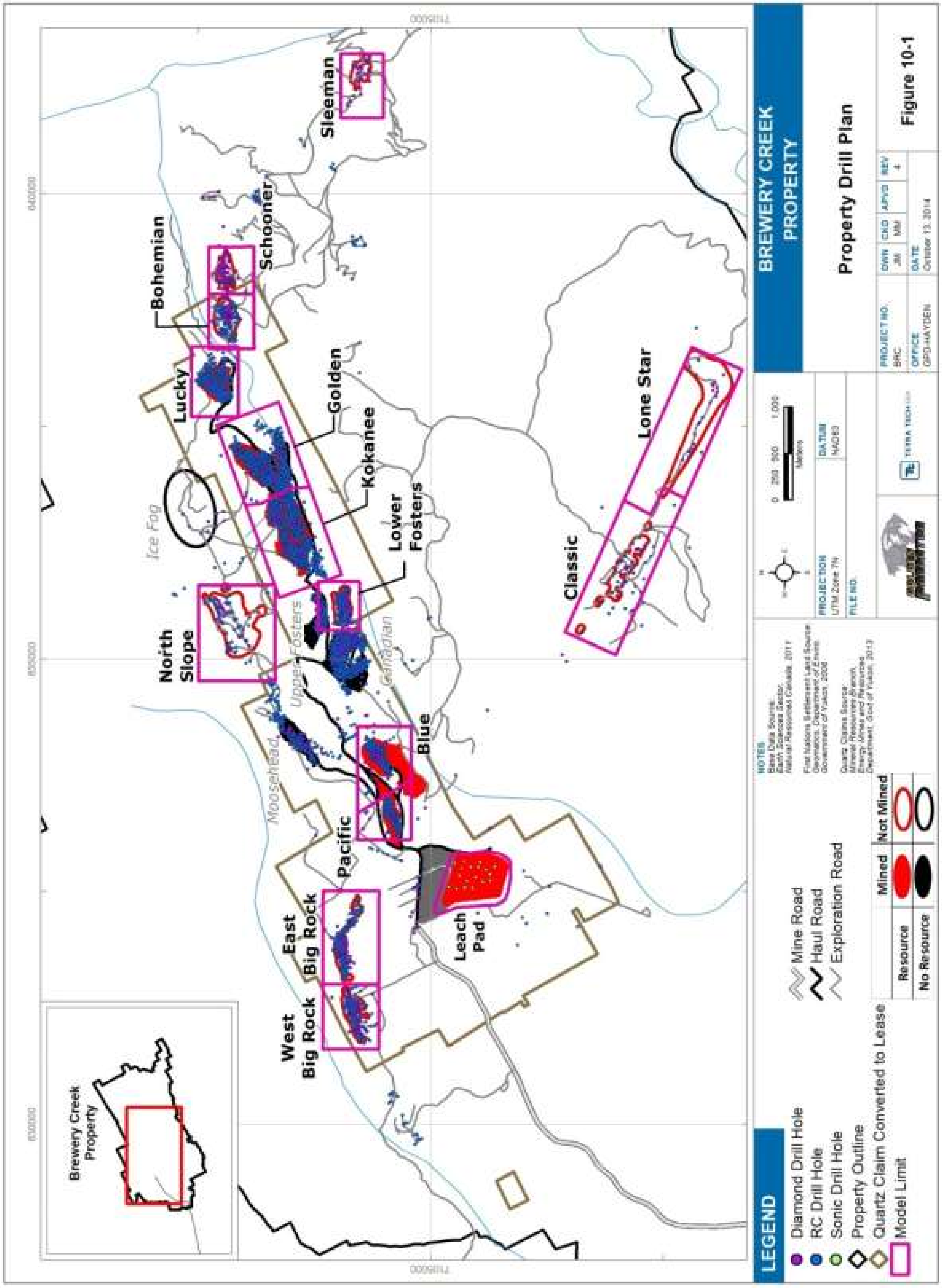
The summary information of the Brewery Creek Project drilling is presented in Table 10-1 and Figure 10.1 below. For drillhole locations by resource area, see Section 14. Golden Predator's drilling was conducted from 2009 through October 2012.

Table 10-1: Summary of Drilling Conducted by Golden Predator, 2009-2012

Drill Series	Year Drilled	Operator*	Drill Type	No. DHs	Total Metres Drilled
BC09	2009	GPY	Core	30	4,981
BC10	2010	GPY	Core	13	2,413
RC10	2010	GPY	RC	16	2,352
BC11	2011	GPY	Core	209	31,054
RC11	2011	GPY	RC	135	24,196
BCS	2011	GPY	Sonic	18	266
BC12	2012	GPY	Core	197	22,227
RC12	2012	GPY	RC	79	9,623
Total				**697**	**97,111**

* Drilling conducted under Golden Predator Corp, Golden Predator Canada Corp., and Americas Bullion Royalty Corp.

Figure 10–1: Property Drilling Map



10.1 Golden Predator Diamond Drilling, 2009

Core drilling in 2009 was completed by Kluane Drilling of Whitehorse, YT, using a KDHT-1000 rig drilling NTW diameter core (56.23 mm). Core was drilled in 3m runs, collected and placed in labeled boxes, and delivered to the on-site core shack at each shift change. Golden Predator staff conducted geotechnical logging, geologic logging and sampling on-site. Downhole surveys were completed with a Reflex-EZ shot tool at 16 m intervals. Collars surveys were completed by a professional land surveyor.

10.2 Golden Predator RC Drilling, 2010

RC drilling in 2010 was conducted by Orbit-Garant of High River, AB, using an 11.4 cm (4 ½ in) diameter bit and interchange system. All sampling was conducted at 1.52 metres (5 ft) intervals and drilling was conducted dry (without added water) until groundwater was encountered. A riffle splitter was used to reduce dry cuttings to a preferred 12.5% split for each interval. A hydraulic rotary splitter was used for sampling if or when wet drilling conditions occurred. Wet sample splits were targeted at the same 12.5% of cuttings as with dry sample splits. Hubco® Sentry II sample bags were used to allow water to escape while retaining fines. Reject material (remaining 87.5%) was also collected for the purpose of future evaluation, assay checks or metallurgical testing.

An on-site geo-technician ensured the splitter was cleaned properly between runs and that sampling was conducted to Golden Predator standards. Additionally, geo-technicians collected a small representative sub sample from each reject bag, washed and placed the representative pieces into plastic chip trays for logging purposes. Detailed geological logs were completed for all holes using a binocular microscope.

Collars were monumented and surveys were completed by a professional land surveyor.

10.3 Golden Predator Diamond Drilling, 2010

Core drilling in 2010 was completed by Peak Drilling of Courtenay, BC. Peak used an EF-50 rig drilling HQ diameter core (63.5 mm). Core was drilled in 3 metres runs, each of which was oriented when possible, and placed appropriate, labeled core boxes. Boxed core was delivered to the on-site core shack, where Golden Predator staff conducted geotechnical logging, geologic logging and sampling. Downhole surveys were completed with a Reflex-EZ shot tool every 16 m. Collars were monumented and surveys were completed by a professional land surveyor.

10.4 Golden Predator RC Drilling, 2011

RC drilling in 2011 was conducted by Boart Longyear of Calgary, AB, and Midnight Sun Drilling Inc. of Whitehorse, YT, using an 11.4 cm (4 ½ in) diameter bit and interchange system. All sampling was conducted at 2 metres intervals and drilling was conducted dry (without added water) until groundwater was encountered. A riffle splitter was used to reduce dry cuttings to a preferred 12.5% split for each interval. A hydraulic rotary splitter was used for sampling if/when wet drilling conditions occurred. Wet sample splits were targeted at the same 12.5% of cuttings as with dry sample splits. Field duplicates were generated by halving the 12.5% split sample material. Tyvek® sample bags were used to allow water to escape while retaining fines.

All drill crew samplers were trained by Golden Predator staff members on sampling. Geo-technicians also collected samples and ensured that proper order was kept during the sampling procedure. The drill crew collected small representative sub-samples from each sample bag, washed them, and inserted them into plastic chip trays for logging purposes. Detailed geological logs were completed for all holes using a binocular microscope. Collars Were Monumented and Surveys were completed by Either a Professional Land Surveyor or by Golden Predator Staff Using a Survey-Grade DGPS instrument.

10.5 Golden Predator Diamond Drilling, 2011

Core drilling in 2011 was conducted by Kluane Drilling or Whitehorse, YT and Peak Drilling of Courtenay, BC. Kluane Drilling used the KDHT-1000 described above, and a KD600, which also drilled NTW core but only with the capacity of 350 metres deep holes. Peak drilling used a Hydracore 2000 and an EF-50. Peak's EF-50 drilled HQ size core (63.5 mm) which had the capacity to drill to 760 metres. Boxed core was delivered to the on-site core shack, where Golden Predator staff conducted geotechnical logging, geologic logging and sampling. Downhole surveys were completed with a Reflex-EZ shot tool at 16 metres intervals. Collars were monumented and surveys were completed by either a professional land surveyor or by Golden Predator staff using a survey-grade DGPS instrument.

10.6 Golden Predator Sonic Drilling, 2011

In July of 2011, Golden Predator completed an 18 hole sonic drilling campaign on the reclaimed leach pad. This program was designed to acquire information on the metallurgical characteristics of heap leach material as well as to collect data for heap leach reactivation. The drilling was completed by Boart-Longyear out of Calgary, AB, using a track mounted sonic drill that was remotely controlled. The machine vertically drilled 10 cm diameter holes by sonically advancing the core barrel followed by casing. Holes were advanced to a maximum depth of 22.86 metres in order to protect the integrity of the liner. Samples were extracted from the core barrel into PVC piping of the same diameter. Sonic sampling occurred at 1.52 metres (5 ft) intervals. Holes w

10.7 Golden Predator Diamond Drilling, 2012

In 2012, drilling was conducted by Kluane Drilling of Whitehorse, YT with a KDHT-1000, and by Matrix Diamond Drilling Inc. of Kimberly, BC with an A5 drill. Boxed core was delivered to the onsite core shack, where Golden Predator staff conducted geotechnical logging, geologic logging and sampling.

Downhole surveys were completed with a Reflex-EZ shot tool at 16m intervals. Collars were monumented and surveys were completed by Golden Predator staff using a survey-grade DGPS instrument.

10.7.1 Golden Predator Metallurgical Drilling, 2012

As part of the 2012 campaign, large diameter PQ core drilling was completed on the property at the West Big Rock, East Big Rock, Moosehead, Lower Fosters, Bohemian, Schooner and Classic areas (BC12-492-510, 512-513, 515-516, 534-535). These large diameter holes were designed for column stack leach testing in an attempt to classify different metallurgical portions of each resource area. Holes were designed by Golden Predator and carried out by on site geological staff. Holes were pre-drilled with RC holes to ensure geologic and oxide boundaries were true as modeled, then followed up with the PQ diameter hole. Core was then selectively sampled for cyanide solubility and column stack leach testing, as intervals were provided by Golden Predator. These intervals were then taken from their respective original core boxes, and placed in separate "composites" for testing. Composites were sent to McClelland for testing. Drilling was conducted solely by Matrix Diamond Drilling from Kamloops, BC with an A5 Zinex drill.

Boxed core was delivered from the drill to the on-site core logging facility where Golden Predator staff conducted recovery analysis and quick logs. Collars were surveyed as above.

10.7.2 Golden Predator Geotechnical Drilling, 2012

Within each of the areas of interest, geotechnical drilling was undertaken for pit design and stability assessment. These holes (BC12-517, 518, 531-533, 536-539, 550-557) were designed by Tetra Tech EBA, and carried out with both Golden Predator and Tetra Tech EBA staff on site. Sampling was done via split tube in 1.52 m sections and geotechnically logged either at the drill (when possible) or back at the on-site core shack. In addition to geotechnical logging, point load testing and specific gravity measurements were taken. Core was oriented for geotechnical purposes (Reflex ACT II RD). Drilling was conducted solely by Matrix Diamond Drilling and core delivery was conducted as discussed in the previous sections.

10.7.3 Golden Predator Groundwater Monitoring Drilling, 2012

Groundwater monitoring holes were drilled outside of the proposed pit boundaries to obtain information on hydraulic conductivity and transmissivity for portions of the geological formations through packer testing. This program consisted of 13 holes, BC12-520-529, 540-542, drilled in the West Big Rock, East Big Rock, Moosehead, Fosters, Schooner and Bohemian zones. Holes were designed by Tetra Tech EBA, and all field work and reporting was conducted by Tetra Tech EBA staff. Drilling was conducted by Kluane Diamond Drilling using a KDHT-1000. These holes were also utilized for geotechnical information when appropriate. Though no oriented data were taken, RMR, RQD and recovery data were all collected through Golden Predator and Tetra Tech EBA staff working on the geotechnical program. In addition, hole BC12-528 and 529 were assayed for metal concentration analysis for exploration purposes. In all cases, holes were transported back to the on-site logging facility for quick logs, geotechnical and sampling by Kluane drilling or Tetra Tech EBA employees.

10.8 Golden Predator RC Exploration Drilling, 2012

RC drilling in 2012 was undertaken initially as a metallurgical characterization and pre-PQ drillhole tool. However, three holes were drilled (RC12-2462-2464) along the BCRT to test mineralization above the previously mined Pacific pit, and to the south west of the previously mined Moosehead pit. The RC drill was then dispatched for metallurgical and pre-PQ work.

Once the metallurgical and pre-PQ holes were drilled, the RC drill was utilized at the Classic zone in order to expand mineralization down dip and along strike of the previously (2011) drilled Classic expansion. RC12-2501-2509 were drilled at Classic and successfully intersected the syenite host rock through the entire length of these holes.

After drilling was completed at the Classic zone, three holes were drilled at the Ice Fog zone, which is an outlying mineralized area located in the northern-central portion of the property and not a part of this report. These holes (RC12-2510 through RC12-2512) were drilled in an effort to follow up on 2011 intersections and historic trench results north of the Golden and Kokanee pits.

After a brief hiatus in the summer, Midnight Sun Drilling Inc. from Whitehorse, Y.T. returned to drill the initial holes in the Lone Star exploration target. Holes RC12-2515 though RC12-2523 and RC12-2527 through RC12-2529 drilled Classic style mineralization in what is mapped as a dextral fault offset of the Classic zone in all holes. In addition to the multi-phase syenite drilled, a number of dykes and a skarn unit were intersected in this portion of the property.

The RC drilling that was conducted by Midnight Sun Drilling used a 3 ½ inch diameter rods with a 10 foot length and a 5 foot sample interval in conjunction with a cross over type bit return. Samples were split into a 12.5% split for each sample interval, and collected in Sunset manufacturing's BVLBL bag for storage and shipment.

All drill crew samplers were trained and overseen by Golden Predator staff members at the drill site. The drill crew collected small representative sub-samples from each bag, washed them then inserted them into plastic chip trays for logging purposes. Detailed geological logs were completed for all holes using a binocular microscope at the on-site logging facility. Golden Predator staff members were on site on each shift (night and day) to enforce sampling methodology. Once the hole was completed it was monumented by Golden Predator staff using a 2x4, then followed up with cementing and surveyed with an RTK survey grade, or DGPS survey grade tool.

10.9 Summary of All Drilling Data

Table 10-2 summarizes all drilling that has been conducted for target areas with reported Mineral Resource Estimates in Section 14 through the Effective Date of the report.

Table 10-2: Summary of Drilling for Resource Estimate Areas

Area	Operator	Core Drilling		RC Drilling		Total Drilling		Percentage of Data
		No. DHs	Metres	No. DHs	Metres	No. DHs	Metres	
Bohemian	Loki	0	0	11	642	11	642	5%
	Viceroy	0	0	96	7,287	96	7,287	55%
	Alexco	3	410	0	0	4	410	3%
	GPY	38	4,263	6	713	44	4,976	37%
	Subtotal	41	4,673	113	8,642	154	13,315	100%
Schooner	Viceroy	0	0	11	1,248	11	1,248	12%
	GPY	81	8,394	5	428	86	8,822	88%
	Subtotal	81	8,394	16	1,676	97	10,070	100%
Fosters (Upper and Lower)	Norex	5	640	3	432	8	1,072	5%
	Loki	13	586	371	14,899	384	15,485	79%
	Viceroy	2	274	9	365	11	639	3%
	GPY	20	1,729	13	692	33	2,421	12%
	Subtotal	40	3,230	396	16,388	436	19,618	100%
West Big Rock	Loki	0	0	25	1,592	25	1,592	11%
	Viceroy	1	141	45	2,412	46	2,553	18%
	GPY	59	6,068	30	3,644	89	9,712	70%
	Subtotal	60	6,209	100	7,648	160	13,857	100%
East Big Rock	Loki	0	0	14	744	14	744	8%
	Viceroy	0	0	80	4,736	80	4,736	50%
	GPY	17	1,925	20	1,981	37	3,906	42%
	Subtotal	17	1,925	114	7,461	131	9,386	100%
Classic	Loki	0	0	11	1,099	11	1,099	8%
	Viceroy	0	0	11	1,634	11	1,634	12%
	Alexco	2	308	0	0	2	308	2%
	GPY	15	3,780	30	6,658	45	10,438	77%
	Subtotal	17	4,088	52	9,391	69	13,478	100%
Lone Star	GPY	17	3,865	12	2,283	29	6,147	100%
Kokanee-Golden	Norex	0	0	4	386	4	386	0.4%
	Loki	29	1,379	482	24,795	511	26,174	55.6%
	Viceroy	14	1,366	377	20,326	391	21,692	42.5%
	GPY	7	1,721	6	933	13	2,653	1.4%
	Subtotal	50	4,466	869	46,440	919	50,905	100.0%

Table 10-2: Summary of Drilling for Resource Estimate Areas

Area	Operator	Core Drilling		RC Drilling		Total Drilling		Percentage of Data
		No. DHs	Metres	No. DHs	Metres	No. DHs	Metres	
Lucky	Loki	3	215	61	3,920	64	4,135	37%
	Viceroy	0	0	102	6,283	102	6,283	56%
	GPY	0	0	6	821	6	821	7%
	Subtotal	3	215	169	11,024	172	11,239	100%
Pacific-Blue	Norex	0	0	0	0	0	-	0%
	Loki	16	776	152	8,091	168	8,867	71%
	Viceroy	7	497	38	1,934	45	2,431	19%
	Spectrum	2	401	0	0	2	401	1%
	Alexco	1	167	0	0	1	167	0%
	GPY	17	2,834	3	416	20	3,250	8%
	Subtotal	43	4675	193	10441	236	15,116	100%
North Slope	Loki	0	0	17	1,032	17	1,032	4%
	Viceroy	2	533	12	1,806	14	2,339	10%
	GPY	30	6,125	79	14,828	109	20,953	86%
	Subtotal	32	6,658	108	17,666	140	24,324	100%
Sleeman	Loki	0	0	7	502	7	502	4%
	GPY	58	10,872	0	0	58	10,872	96%
	Subtotal	58	10,872	7	502	65	11,374	100%
GPY Only		359	51,576	210	33,397	569	84,971	43%
Total		459	59,270	2,149	139,562	2,608	198,829	

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Collection Methods

11.1.1 Historical Sampling by Norex, 1989

Information for the Norex sampling preparation and analysis program were not available to Tetra Tech EBA at the time of reporting. A total of 7 diamond drillholes and 5 RC holes from this campaign were drilled within the Fosters-Canadian area discussed in this report. The material surrounding the significant mineralized intervals of these holes has now been mined and these holes do not contribute to the current mineral resources found on the property and their sampling methodology is considered irrelevant to this report.

11.1.2 Historical Sampling by Loki and Viceroy, 1990-1999

The details of core and RC drill sample preparation, QA/QC, analysis and security procedures prior to 2004 are generally absent in the project files. Based on Viceroy Drill and sample logs, samples were logged and collected on continuous 2 metre intervals downhole and submitted to the laboratory for Au and metallurgical test work. During production, much of the analysis was completed in the on-site laboratory with periodic verification sampling sent offsite to certified labs, as indicated in Table 11.1, below.

During this period approximately 63,300 samples were collected and analyzed by five different laboratories. Table 11-1 provides details of the information currently available with respect to operator, sample quantities, laboratories and analytical analysis.

Table 11-1: Historical Analytical Laboratories and Methods

Period	Operator	App. Samples	Laboratory	Analytical Method
1989	Norex	1,300	Norex	Hot Aqua Regia Digestion with AA Analysis
1990-1992	Loki	18,000	Acme	Hot Aqua Regia Digestion with AA Analysis
1993-1995	Loki	18,000	Terramin	30g Fire Assay with AA Finish
1996-1999	Viceroy	29,000	Brewery Creek Mine	30g Fire Assay with AA Finish
2004	Spectrum	382	ALS Chemex	30g Fire Assay with AA Finish
2006	Alexco	783	ALS Chemex	30g Fire Assay with AA Finish

11.1.3 Historical Sampling by Spectrum and Alexco, 2004-2006

Sampling procedures in 2004 and 2006 were as follows. The geologist laid out each sample by marking the start and end of the sample in red china marker on the core. The first part of the sample tag was stapled onto the core box at the start of the sample. If the next sample was a standard, blank or duplicate, that sample tag was stapled onto the box next to the previous tag. The second part of the sample tag was then placed into a plastic sample bag and the number written in marker onto the bag. The core was then transferred to the core cutting area.

The core was cut in half longitudinally using a 14” core saw. The technician placed one half of the core into the sample bag with the corresponding sample tag stapled on the core box. When a second tag was beside the first tag, the technician placed either the blank material or standard material into the next sample bag, based on what was written on the sample tag. When the second tag called for a duplicate, the technician placed an empty sample bag with a sample tag included into the previous sample. Each bag was then closed and secured with a zap strap.

Once twenty sample bags were collected (a complete batch), each batch was placed into rice sacks and labelled with the batch number, bag number, sample numbers within batch, and ALS Chemex’s North Vancouver address. Each rice bag was then taped shut and secured with a zap strap. Twice a week, the rice bags were delivered to Mayo and placed on the Kluane Transport Ltd. truck for Whitehorse, where it was shipped to Northwest Freight Systems for transportation to ALS Chemex (ALS) in North Vancouver.

11.1.4 Golden Predator Core Sampling, 2009-2012

Sampling procedures used from 2009 through 2012 were as follows. Core was oriented (when applicable), retrieved from the barrel, cleaned, placed into boxes and transported to the on-site core logging facility by either drilling crew or Golden Predator staff. Upon arrival at the logging facility, core was logged and tagged for sample breaks. Sample tags, labeled with numeric sample ID, were then attached to core boxes at appropriate sample break points. A preferred sample interval of 2 metres was used whenever possible, but varying sample intervals were used to honor lithologic contacts, significant structural features, alteration and mineralized intervals. Prior to sampling, geotechnical and oriented structural measurements were recorded, specific gravity of select lithologic units was measured and the core was photographed.

A diamond saw located at the on-site core logging facility was used to sample halved core; field duplicates were generated using ¼ core samples. Samples were placed in pre-labeled poly bags and grouped into batches within labeled, sealed/secured rice sacks in preparation for shipment to the lab. Unsampled ½ core was returned to the original core box for storage on-site.

Batch sizes in 2009 and 2010 consisted of 36 samples, including QA/QC SRM (Au standards and blanks) and field duplicates. Mid-season 2011, batches were increased from a 36 sample count to include all samples for each drillhole. This method of whole-hole batch sizing has been used consistently since the 2011 change, and throughout the 2012 program.

11.1.5 Golden Predator RC Sampling, 2010-2011

In 2010, RC drill samples were collected from an 11.4 cm (4 ½ in) diameter drillhole with a uniform 1.52 metres (5 ft) sample interval. Dry cuttings were funneled from the cyclone through a three-tier Jones (riffle) splitter, setup to gather 12.5% of the returned material. A hydraulic rotary splitter was used for sampling if/when wet drilling occurred. Wet sample splits were targeted at the same 12.5% of cuttings as with dry sample splits. All samples were contained in pre-labeled Hubco® Sentry II bag, which allows for water drainage while retaining fines.

Each sample was identified using a blind assay tag number placed in the sample bag. The corresponding sample number was also written on the sample bag. Bags were sealed and collected at the drill, placed into pre-labeled rice bags and were transported to the logging area by either the drillers or Golden Predator staff. Sample batches of 36 were accumulated for shipment. Each batch of 36 samples included, one blank, one standard reference material, and one duplicate. Field duplicates were generated by splitting the remaining (87.5%) sample material.

Sampling in 2011 was collected over 2m intervals from an 11.4 cm (4 ½ in) diameter hole. Dry cuttings were funneled from the cyclone through a three-tier Jones (riffle) splitter, setup to gather 12.5% of the returned material. A hydraulic rotary splitter was used for sampling if/when wet drilling occurred. Wet sample splits were targeted at the same 12.5% of cuttings as with dry sample splits. All samples were contained in pre-labeled Tyvek® bag, which allows for water drainage while retaining fines.

Each sample was identified using a blind assay tag number placed in the sample bag. The corresponding sample number was also written on the sample bag. Bags were sealed and collected at the drill, placed into pre-labeled rice bags and were transported to the logging area by either the drillers or Golden Predator staff. Field duplicates were generated by halving the 12.5% sample split with a box splitter. Entire holes were placed in apple crates and shipped as individual batches, which included inserted blank and standard reference material.

11.1.6 Golden Predator RC Sampling, 2012

In 2012, RC samples were collected over 1.52m intervals from an 8.89 cm (3 ½ in) diameter hole. Dry cuttings were funneled from the cyclone through the three-tier Jones (riffle) splitter, setup to gather 12.5% of the returned material. A hydraulic rotary splitter was used for sampling if/when wet drilling occurred. Wet samples were targeted at the same 12.5% of cuttings as with dry sample splits. All samples were contained in Sunset Manufacturing BVLBL bags, which were pre-labeled. These bags allowed for drainage of excess water while retaining fines.

Each sample was identified using a blind assay tag number placed in the sample bag. The corresponding sample number was also written on the sample bag. Bags were sealed and collected at the drill, and transported to the logging area by either the drillers or Golden Predator staff. After a period of time for draining of excess water, bags were placed in pre-labeled sample bins (apple crates) with a corresponding batch label (batched by hole). Field duplicates were generated by halving the 12.5% sample split with a box splitter. Entire holes were placed in apple crates and shipped as individual batches which included blank, standard reference material, and the aforementioned duplicates.

11.2 Sample Processing and Security Measures

During a site visit in March, 2012, Tetra Tech EBA reviewed the sample collection and processing protocol being implemented on site. The facilities in place at the time consisted of dedicated core receiving/logging, cutting and processing areas as depicted in Figure 11–1 through Figure 11–3 below. Security and control on sample handing is measured through the process and is described in subsequent sections.

RMI conducted a similar review of sample collection during a site visit in mid-October 2012.

During Gustavson's site visit, no drilling or sampling were being performed, so sampling security measures were not observed directly.

All sampling was conducted under the supervision of a Golden Predator project geologist and the chain of custody from the drill to the sample preparation and logging facility was monitored by the project geologist. Samples were shipped to the lab by qualified couriers or Golden Predator personnel under security-tagged bags with independent identification numbers.

Figure 11–1: Core Logging Facility, March 2012

Figure 11–2: Core and Sample Processing Facility, March 2012

Figure 11–3: Onsite Core Cutting Equipment, March 2012

11.3 Sample Analytical Methods

11.3.1 Historical Analytical Methods by Norex, 1989

Sampling methods used by Norex are unknown.

11.3.2 Historical Analytical Methods by Loki, and Viceroy, 1990 – 1999

Drill logs and laboratory certificates recovered from Loki/Viceroy drilling campaigns indicate that analysis was conducted using aqua regia digestion with atomic absorption finish during the years 1990 through 1992 at ACME laboratories. The method was changed to 30 g fire assay using atomic absorption finish during Loki/Viceroy drilling between the years 1993-1999 at Terramin Labs and the on-site laboratory.

Some of Loki and Viceroy's samples were assayed at ALS, though actual methods used are not known.

11.3.3 Historical Analytical Methods by Spectrum, 2004

The analytical methods used by ALS for the Spectrum 2004 drill samples were as follows. ALS sample preparation (Prep 31) procedure, which involves finely crushing the entire sample to better than 70% -2 mm, splitting off up to 250 g and pulverizing the split to better than 85% passing 75 micron. Gold was analyzed by ALS procedure Au-AA-25, a fire assay – atomic absorption finish method. Samples were also assayed for 34 metals by ME-ICP41, an aqua Regia digestion and analysis by inductively coupled plasma-atomic emission spectroscopy (ICP-ES).

11.3.4 Historical Analytical Methods by Alexco, 2006

The analytical methods used by ALS for the 2006 drill samples were as follows. ALS sample preparation (Prep 31), then assayed for gold by Au-AA25.

Analysis for an additional 27 elements was completed using ALS method ME-ICP61, a hot four-acid digestion and analysis by ICP-ES.

11.3.5 Golden Predator's Analytical Methods, 2009

ACME Analytical Laboratories of Vancouver, B.C. performed all sample preparation and analyses. ACME Analytical Laboratory is certified by ISO 9001:2008 FM 63007.

Core samples were logged and sampled at the project site under the supervision of the project geologist and then expedited in sealed bags to Whitehorse where they were shipped via common carrier to Vancouver. After being received and logged in at the laboratory, a 2 kg split of core was dried then crushed to 80% -10 mesh. A 250 g split was then pulverized to 85% -200 mesh (Sample Preparation Method R200-250).

A 15 g split of each sample was analyzed by ICP-MS after Aqua Regia digestion to yield a 37 element scan (Method 1F01). All samples yielding greater than 500 ppb gold then underwent a 30 g fire assay with an ICP-ES finish (Method G6). QA/QC procedures followed for the diamond drilling program include submittal of assay standards for analysis approximately every 30 samples as well as a blank and a duplicate sample of quarter core at approximately the same frequency.

11.3.6 Golden Predator's Analytical Methods, 2010

All drill core and RC chips samples in 2010 were received at the ALS Chemex sample prep facility in Whitehorse, YT and analyzed by ALS Chemex in Vancouver, BC. ALS Chemex Laboratory in Vancouver Canada is certified by ISO 9001:2008 and ISO/IEC 17025:2005. Identical procedures were used for both RC and core samples. Samples were prepared in accordance with Prep 31 requirements. Samples were assayed for gold by Au-AA23, with reporting limits of 0.005 to 10 ppm. Samples were also analyzed for 35 elements by ME-ICP41.

11.3.7 Golden Predator’s Analytical Methods, 2011

Drill core and RC samples in 2011 were received at either ALS Minerals Whitehorse, YT sample prep facility or at one of ACME Laboratories Dawson City, YT or Whitehorse, YT sample prep facilities. Sample analysis was conducted by either ACME Laboratories, Vancouver, BC or by ALS Minerals, Vancouver, BC or Reno, NV.

Samples sent to ACME were prepared using Method R200-250. ACME assayed for gold by Method G6, 0.005g/t detection limit, 10 ppm upper limit, fire assay of 30g Atomic Absorption finish (Automatic Gravimetric Overlimit); and by Analytical Method Code 7TD1 for silver only (2g/t detection limit), which consists of hot 4-Acid digestion of 1 g minimum pulp for sulfide and silicate ores followed by ICP-ES analysis.

Samples submitted to ALS are prepared using method Prep 31, followed by gold assay by Au-AA23, and for 35 elements by ME-ICP41.

11.3.8 Golden Predator’s Analytical Methods, 2012

Golden Predator’s 2012 samples were prepared by Prep 31, as described in Section 11.2.3, followed by gold assay by Au-AA23 as described in Section 11.2.6. Some samples were analyzed for multi-elements by ME-ICP41, as described in Section 11.2.6. All samples that returned gold grades in excess of 200 ppb (0.2 ppm) were re-analyzed by cyanide leach and gold preg-robbing methods (Au-AA31 and Au-AA31a).

Part way through their 2012 drilling campaign, Golden Predator ran cyanide leach analyses (AuAA13) on all intrusive samples where the initial fire assay grade was in excess of 0.2 g/t.

11.4 Analytical Quality Assurance and Quality Control

11.4.1 RMI Review of Database

RMI obtained Excel spreadsheets from Golden Predator that contained various 2012 QA/QC data for the Bohemian, Schooner, Fosters, West Big Rock, East Big Rock, Classic, and Lone Star deposits. RMI notes that while the assay data were completed in 2012 some of the data represents late 2011 drillholes.

Table 11-2 summarizes the number of samples that were analyzed in 2012 by resource area. The "Assay" column refers to drill core intervals that were sampled. The "SRM" column refers to standard reference material or "standards". The "Duplicate" column refers to paired ¼ core and split RC samples that produce an "original" and a "duplicate" sample from the same interval. The "LPl" and "LRj" columns refer to additional samples that were prepared by ALS Chemex from pulps and coarse rejects.

Table 11-2: List of QA/QC control and Standard Reference Materials from 2012

Area	Number of Samples					
	Assays	Blanks	SRM’s	Duplicates	LPI	LRj
Bohemian	**324**	**9**	**10**	**7**	**2**	**4**
Schooner	1,192	38	42	34	12	6
Fosters	620	19	17	12	4	3
West Big Rock	2,950	116	104	67	24	25
East Big Rock	1,209	34	39	21	18	15

Table 11-2: List of QA/QC control and Standard Reference Materials from 2012

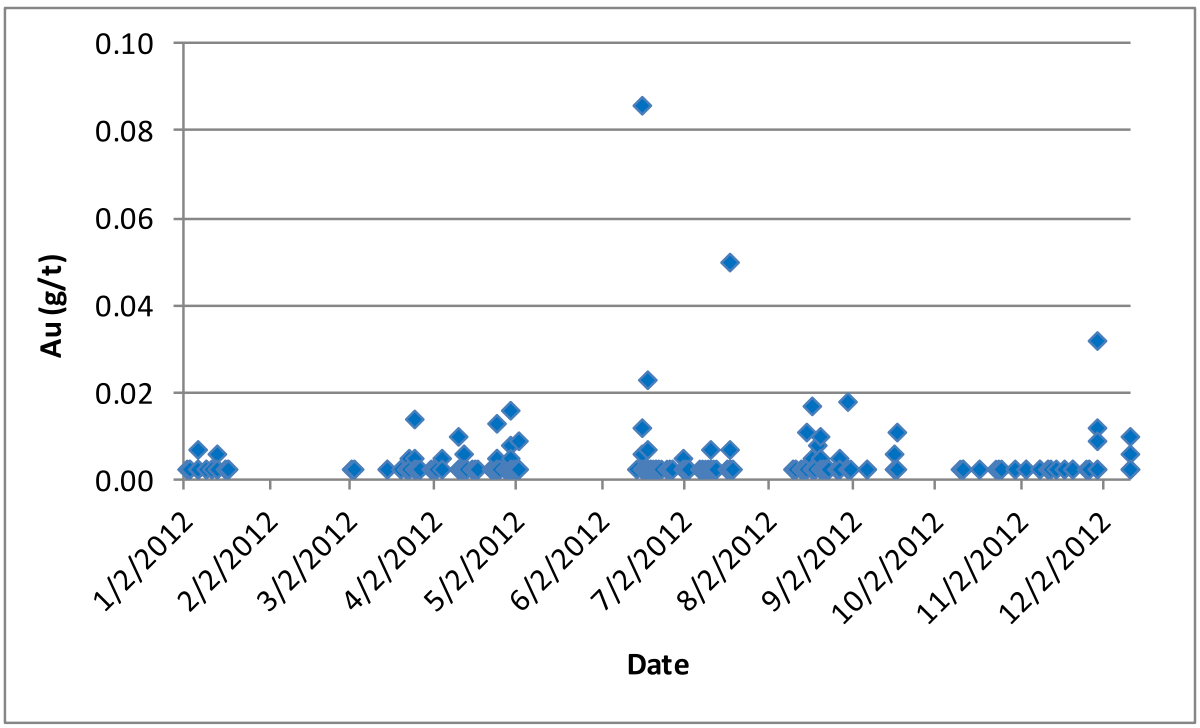
Area	Number of Samples					
	Assays	Blanks	SRM's	Duplicates	LPI	LRj
Classic	3,916	108	109	105	57	55
Lone Star	3,374	93	89	57	62	66
Total	13,585	417	410	303	179	174

Golden Predator submitted blanks at a frequency of about one blank for every 33 regular samples. SRM's were submitted at approximately the same frequency as blanks while duplicate samples were generated about every 45 samples.

11.4.1.1 2012 Blank Performance

For their 2012 drilling campaign Golden Predator purchased decorative rock (quartz) from a local garden supply store. Figure 11–4 graphs blank gold grades that were assayed by ALS Chemex as a function of time.

Figure 11–4: Performance of 2012 Blanks



The detection limit for the Chemex fire assay data for 2012 was 0.005 g/t. Golden Predator has used a 0.01 g/t threshold for flagging potential assay failures. Most of the assayed blanks in 2012 fall below the 0.01 threshold and nearly all of the blank samples were less than 10 times the detection limit. There does appear to be occasional trace amounts of gold in some of the decorative stone that was used as barren material but in general it does provide a reasonable measure of how well the lab is performing.

11.4.1.2 2012 SRM Performance

In 2012 six standards were submitted at a frequency of about 1 SRM for every 33 regular samples. The certified standards were prepared and purchased from CDN Resource Laboratories Ltd. out of Langley, B.C. Two of the standards used in 2012 (CDN-GS-5J and CDN-GS-P3C) were seldom used. Table 11-3 summarizes the commercial standards that were used in 2012.

Table 11-3: Summary of SRM's Submitted in 2012

SRM	Expected Au Value (g/t)	2 Standard Deviations	Number of SRM's Submitted
CDN-GS-P3C	0.263	0.020	1
CDN-GS-P3B	0.409	0.042	123
CDN-GS-P4A	0.438	0.032	77
CDN-GS-1P5D	1.470	0.150	154
CDN-GS-5G	4.770	0.400	54
CDN-GS-5J	4.960	0.420	2
Total	n/a	n/a	411

Figure 11–5 through Figure 11–8 graphs the performance of standards CDN-GS-1P5D, CDN-GS-P3B, CDN-GS-P4A, and CDN-GS-5G, respectively. The SRM graphs show the ALS Chemex result as a function of time and all contain ±2 and ±3 standard deviation lines.

Figure 11–5: Performance of 2012 SRM CDN-GS-1P5D

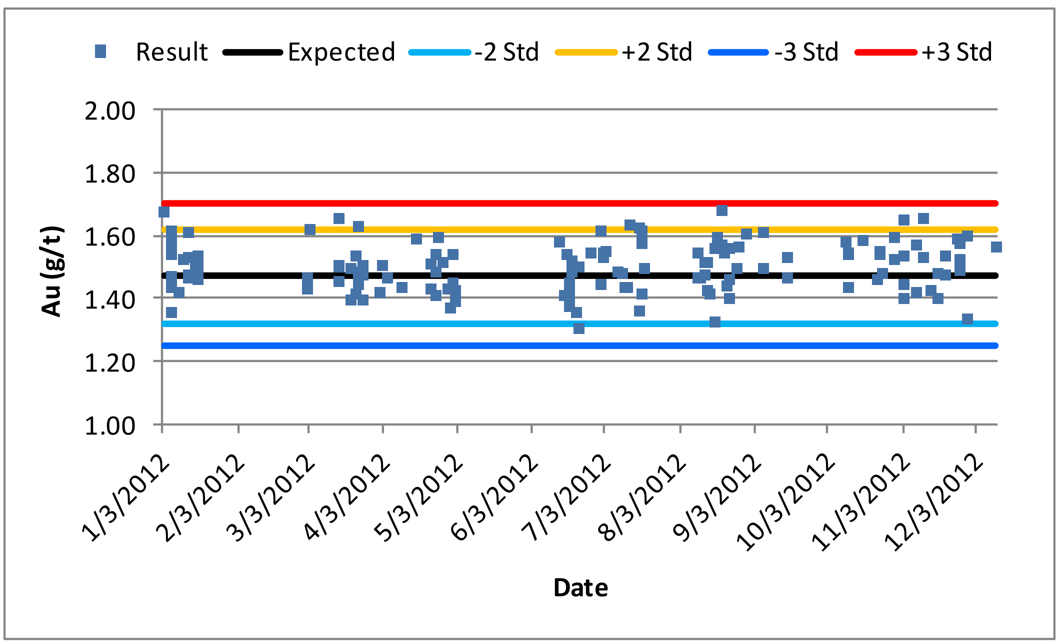


Figure 11–6: Performance of 2012 SRM CDN-GS-P3B

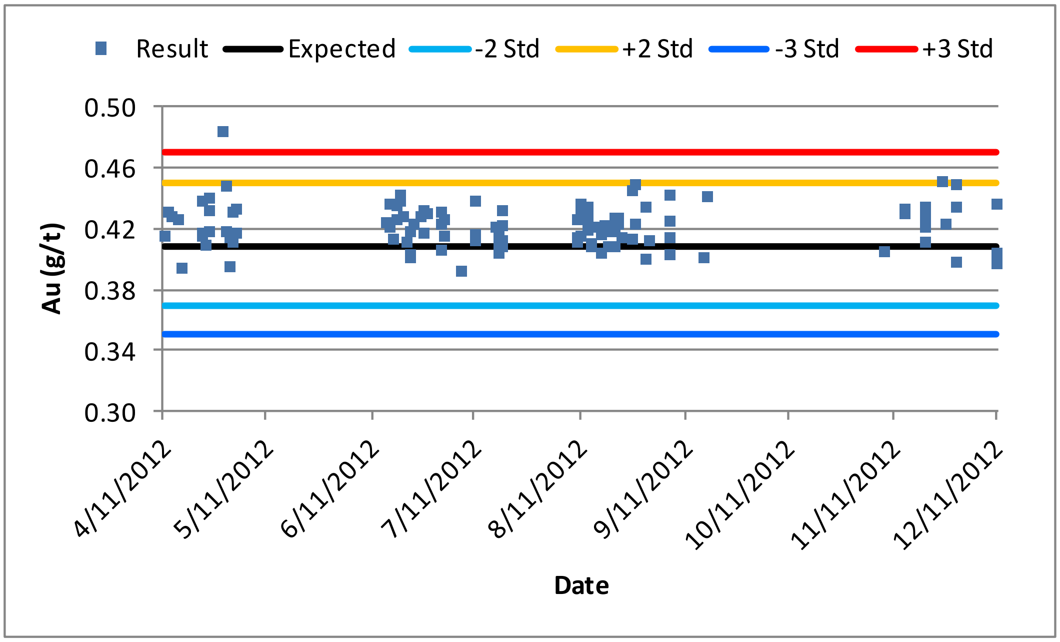


Figure 11–7: Performance of 2012 SRM CDN-GS-P4A

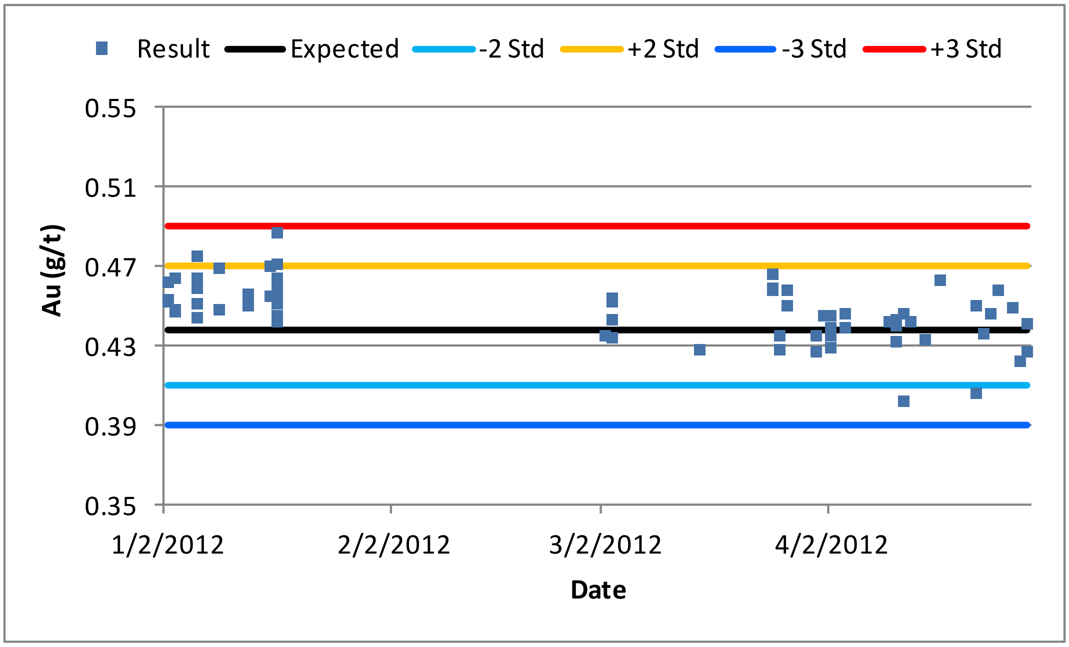
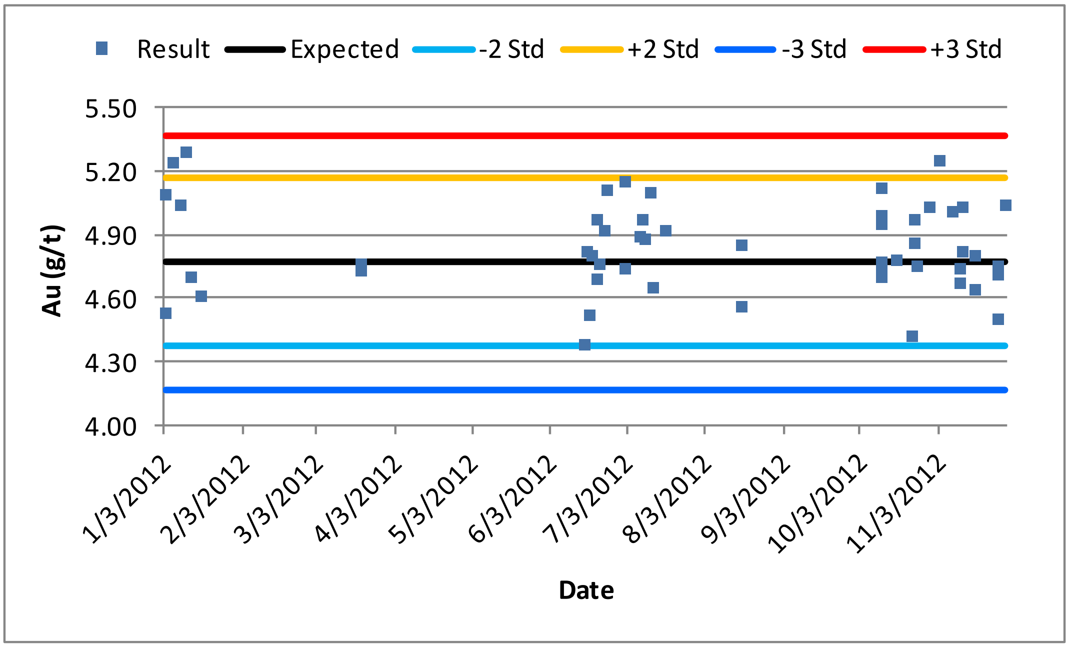


Figure 11–8: Performance of 2012 SRM CDN-GS-5G



In general most of the SRM's that were assayed by ALS Chemex in 2012 fell comfortably within ±2 standard deviations. Only two samples fell outside of ±3 standard deviations and one of those may have been a blank that was inadvertently labeled as a standard and the other was just slightly outside of +3 standard deviations.

11.4.1.3 2012 Duplicate Sample Performance

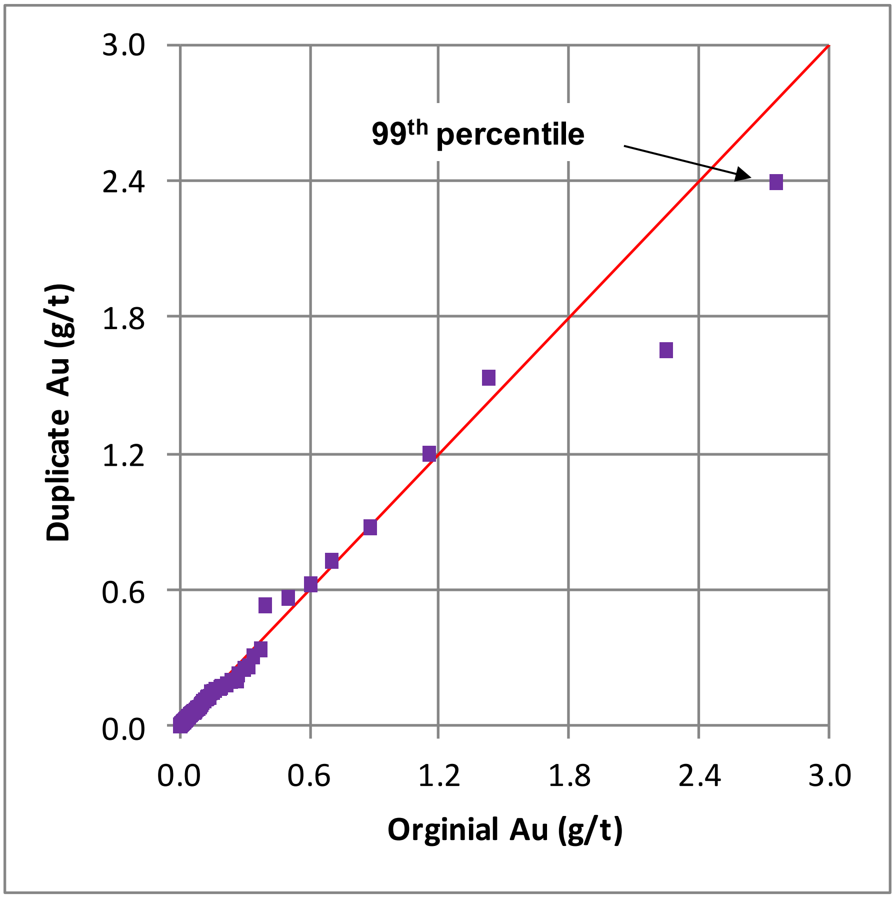
As mentioned above, duplicate samples were prepared at a frequency of about one ¼ core duplicate or RC split for every 45 regular samples. Table 11-4 summarizes basic statistics for the original and duplicate samples.

Table 11-4: 2012 Duplicate Sample Statistics

Parameter	Count	Meters	Min Au (g/t)	Max Au (g/t)	Mean Au (g/t)	Std. Dev.	CV
Original Sample	303	542.13	0.002	6.000	0.178	0.545	3.057
Duplicate Sample	303	542.13	0.002	6.010	0.172	0.527	3.058

RMI notes that the mean gold grade for the original samples was 3% higher than the duplicate sample. In RMI's opinion this is not a material issue given the inherent variability of gold deposits. Figure 11–9 is a quantile-quantile (QQ) plot that compares the original sample (X-axis) with the duplicate sample (Y-axis).

Figure 11–9: 2012 Duplicate Sample QQ Plot



The data shown in Figure 11–9 shows a reasonable comparison below approximately 1.5 g/t or about 97 percent of the data. The 98th and 99th percentile data points show a definite bias towards the original sample.

11.4.2 Gustavson Review of Database

11.4.2.1 Standards

Available standard samples and results as provided from Golden Predator are summarized in Table 11-5. As shown on Table 11-5, of the 1,746 standard samples from 2004 through 2012, 6% of the standards exceeded the acceptance criteria, which was the certified standard result, plus or minus 3-times the certified standard deviation results. Gustavson notes that approximately two-thirds of the samples were within range and below the reported standard mean value, potentially suggesting corresponding gold assay are under-reported, rather than over-reported. Gustavson concludes that the available standard results are acceptable.

Table 11-5: Summary of Available Standard Sample Results

Analysis Date	Standard Name	Upper Range	Lower Range	No. Samples	Total Samples Outside Range	Samples Over Range	Samples Under Range
2004	STD-B (Note 1)	1.36	1.15	11	2	1	1

Table 11-5: Summary of Available Standard Sample Results

Analysis Date	Standard Name	Upper Range	Lower Range	No. Samples	Total Samples Outside Range	Samples Over Range	Samples Under Range
2004	STD-A (Note 1)	6.3	5.2	12	1	1	0
2006	Std-PM182 (Note 1)	1.36	1.15	21	3	0	3
2006	Std-PM907 (Note 1)	6.25	5.17	24	2	0	2
2011	SRM_GSP2	0.24	0.18	20	0	0	0
2011	SRM_GS1F	1.36	0.96	17	2	1	1
2011	SRM_GS2E	1.73	1.31	3	0	0	0
2011	SRM_GS1P5C	1.75	1.37	42	3	0	3
2011	SRM_GS4B	4.18	3.37	10	0	0	0
2011	SRM_SN50	9.19	8.11	13	0	0	0
2012	SRM_GSp3B	0.47	0.35	131	2	1	1
2012	SRM_GS5J	5.59	4.33	4	1	0	1
2012	SRM_GSp3C	None Provided		1	--	--	--
2009, 2010	Std-NR	None Provided		28	--	--	--
2009, 2010, 2011	SRM_GS1D	1.2	0.9	56	11	10	1
2009, 2010, 2011	SRM_GS10C	10.69	8.73	32	1	1	0
2010, 2011	SRM_CM-7	0.49	0.36	8	0	0	0
2010, 2011	SRM_OXE74	0.67	0.56	45	0	0	0
2010, 2011	SRM_OXH66	1.38	1.19	29	0	0	1
2010, 2011	SRM_GS2F	2.52	1.8	11	0	0	0
2010, 2011	SRM_CGS-21			13	0	0	0
2011, 2012	SRM_GSP4A	0.49	0.39	410	34	7	27
2011, 2012	SRM_GS1P5D	1.7	1.25	457	25	3	22
2011, 2012	SRM_GS5G	5.37	4.17	348	23	14	9
Total Number of Samples				**1,746**	**110**	**36**	**72**
Percentage of Samples					**6%**	**35%**	**65%**

Note 1 – Results of identified standard samples were taken from Tetra Tech EBA (2012)

11.4.2.2 Blanks

A decorative stone (reddish shale) purchased from Home Hardware in Whitehorse, YT, was used as the blank material for both the 2004 and 2006 drill programs. Blank material for the 2009 program was sourced from an on-site sandstone outcrop located near the core storage area. This material was found to be unsuitable as it contains trace Au values and was not used in future programs. Blank material used for the 2010 to 2012 programs was a bull-quartz landscaping product called "Garden Quartz", packaged by Hillview Products of Barrie, ON.

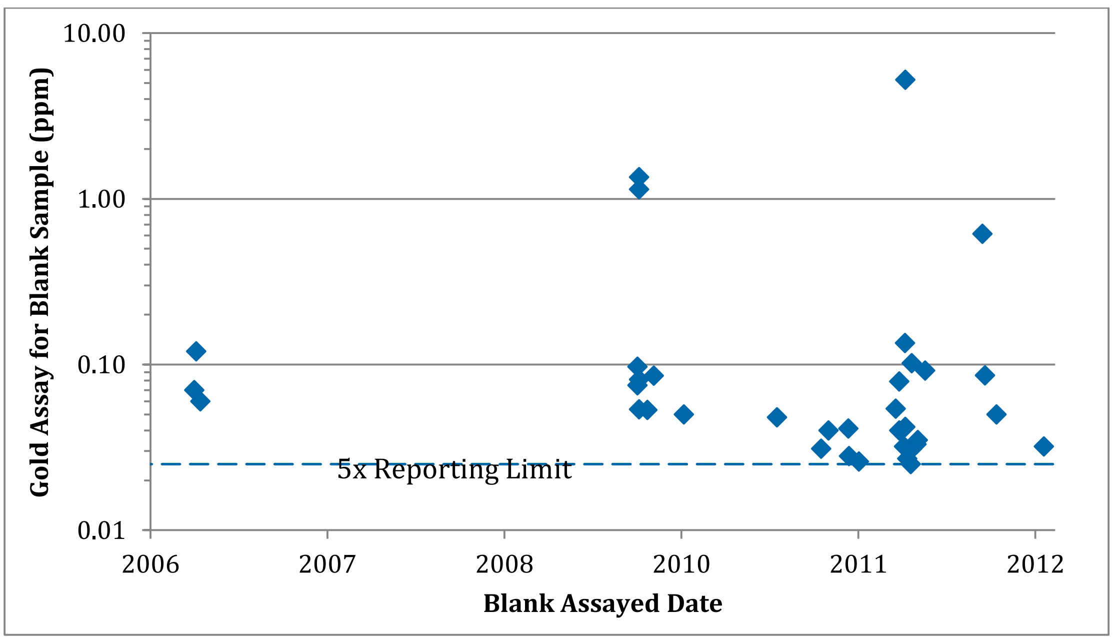
Blank sample results are shown on Table 11-6. Gustavson considered those blank sample detections at levels less than 5-times the reporting limit (RL) to be acceptable. As shown in Table 11-6, of the 1,776 blank samples, 36 blank samples exceeded the 5-times reporting limit acceptance criterion. Gustavson concludes that the blank sample results are acceptable.

Table 11-6: Summary of Blank Sample Results

Gold Assay Method	Year	Reporting Limit (ppm)	Number of Blank Samples	Detections > 5*RL	% > 5*RL
ALS_Au-AA25	2004-2006	0.01	51	3	6%
ACM_1F	2009-2010	0.01	74	6	8%
ACM_G6	2009-2012	0.005	174	5	3%
ALS_Au-AA23	2010-2012	0.005	1,477	22	1%
Total Number of Samples			1,776	36	2%

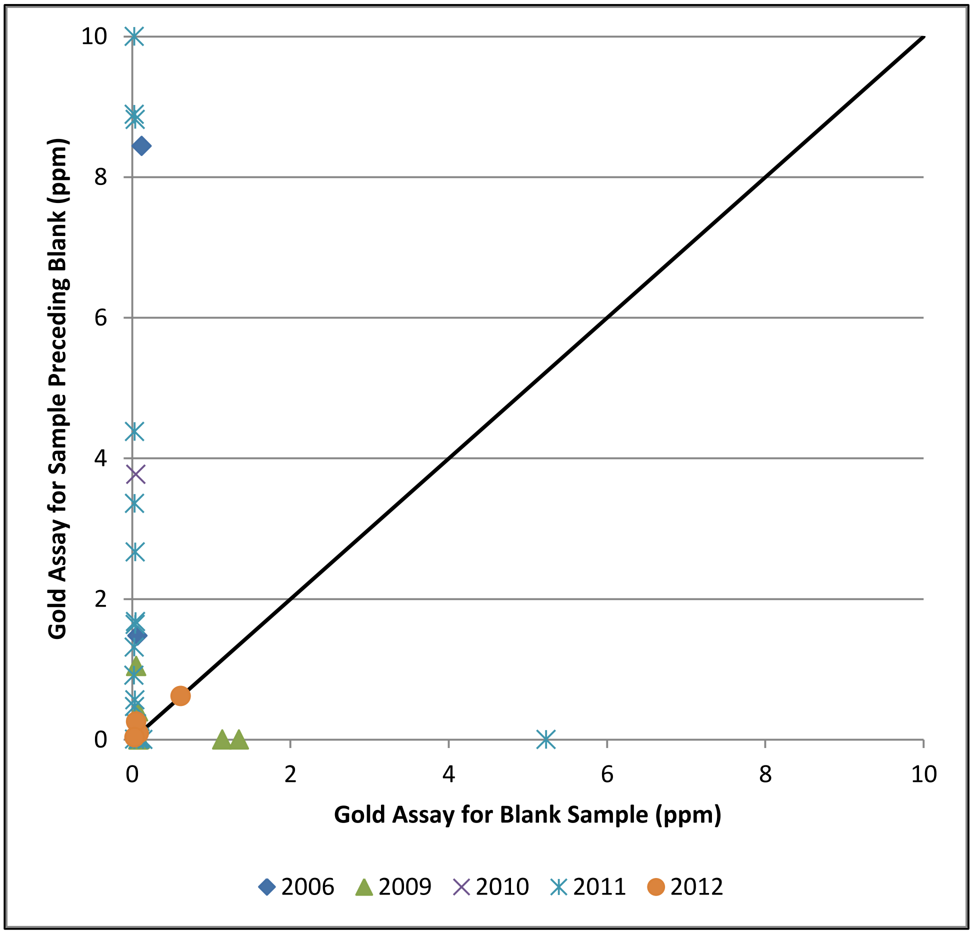
Those blank samples containing gold at levels greater than 5-times the reporting limit are plotted on Figure 11–10. Gustavson notes that all of the blanks with detections greater than 5-times the reporting limit were noted in samples assayed in the Acme laboratory by the G6 Method or in the ALS laboratory by the AA23 Method: both with reporting limits of 0.005 ppm.

Figure 11–10: Blank Gold Assay Data, if Detected 5-Times above RL



Those blank samples with gold detections greater than 5-times the reporting limit are shown on Figure 11–11, along with the gold assay result of the sample preceding the blank. This was done to determine whether the gold detections in blanks are a result of carry-over, that is, high levels of gold from the preceding sample carrying over into the blank.

Figure 11–11: Blank Gold Assay Data, if Detected 5-Times above RL



Combined, Figure 11–11 and Figure 11–12 show that blank detections above the acceptance criteria may be due to a combination of two factors:

- Blank samples potentially contain gold, as evidenced by high detections in the blank sample that are not preceded by sample containing comparably high levels of gold. Gustavson notes that this phenomenon is rare, and as such, concludes that the existing blank samples are acceptable for future use.
- Carry-over of gold from a sample containing high gold is occurring, as evidenced by high gold detections in samples preceding blank sample that exceed the acceptance criteria. Gustavson suggests that Golden Predator discuss employing more robust QA/QC practices at the laboratory, in an effort to reduce the potential for gold carry-overs.

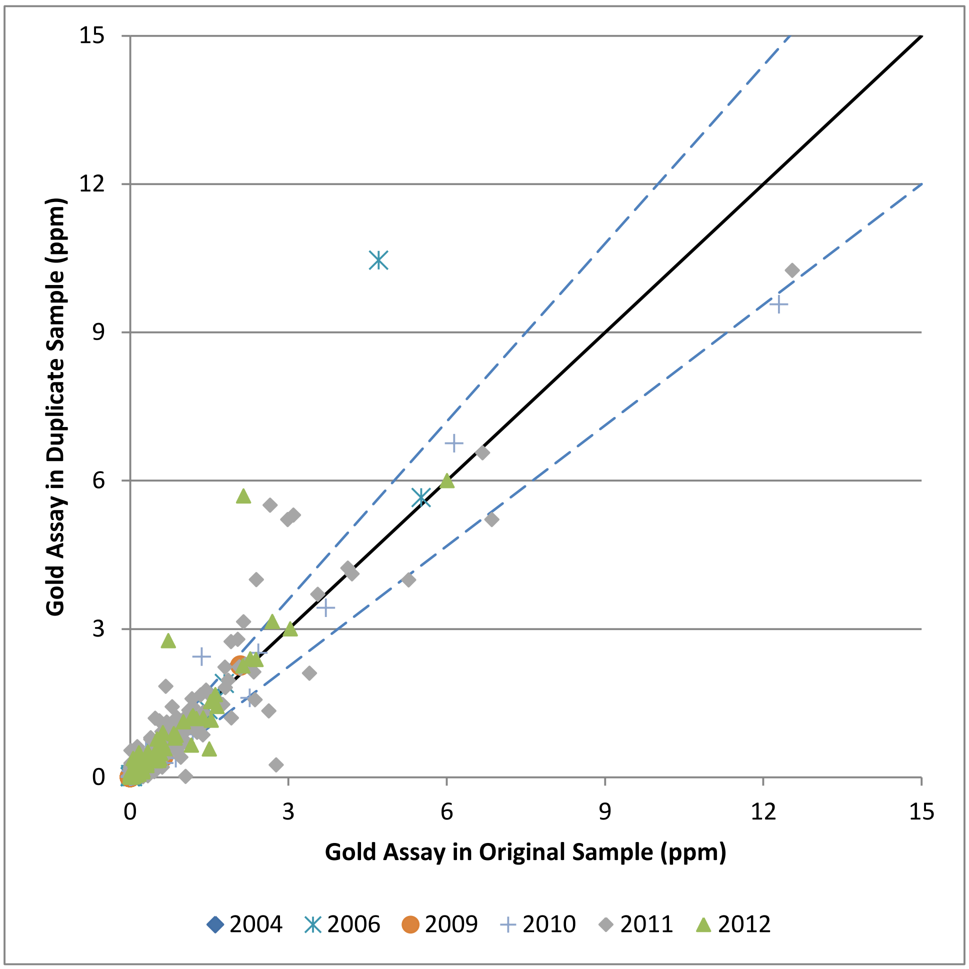
11.4.2.3 Duplicates

A total of 1,627 duplicate samples were provided to Gustavson from 2004 through 2012, as shown in Table 11-7. A plot showing original and duplicate sample results are provided on Figure 11-12 shows acceptable agreement between original and duplicate sample throughout the years when duplicate data are available.

Table 11-7: Summary of Duplicate Samples

Year Analyzed	Duplicate Samples
2004	13
2006	38
2009	44
2010	103
2011	1071
2012	358
Total	1627

Figure 11–12: Duplicate Sample Results



11.5 QAQC Compliance Statements

11.5.1 RMI Statement

During RMI's site visit in October of 2012, sampling and security procedures were reviewed with Golden Predator's geologic staff. RMI did discover that some of the logged lithologic and oxidation data were not being fully transferred from paper drill logs to the electronic database. It is RMI's understanding that Golden Predator has revamped their recordation procedures so that the data is now being correctly captured.

It is RMI's opinion that the sample preparation, security, and analytical procedures implemented by Golden Predator are reasonable and suitable for technical use under NI 43-101 disclosure requirements. Pre-Golden Predator data were verified by comparing the distribution of older RC sample data with Golden Predator diamond core results, which have been validated by suitable QA/QC protocols (submission of blanks, certified standard reference samples, and duplicate samples).

11.5.2 Gustavson Statement

Based on Gustavson's assessment of sample collection, analytical, security, and QA/QC procedures, Gustavson concludes that the data are adequate for supporting an NI 43 101 resource estimate.

11.5.3 Tetra Tech EBA Statement

Tetra Tech EBA has documented a review of sample collection, analysis, security and QAQC in a previous report titled "Updated Mineral Resource Estimate for the Brewery Creek Property, Yukon Territory, Canada" dated October 2, 2013.

A first hand review of the procedures was completed while on site in March of 2012. The sampling procedure being implemented by Golden Predator at that time and the data being used in the geological database has been subjected to a rigorous QAQC regime. A thorough screening protocol was being adhered to. The data is considered to be valid and suitable for technical use under NI 43-101 disclosure requirements.

12.0 DATA VERIFICATION

12.1 Tetra Tech EBA Verification of Historical Data

Data verification was completed by Tetra Tech EBA in the recent technical disclosure report, with Effective Date of March 11, 2012. This work remains relevant and is summarized below.

Physical drill core and RC chip sample records for historical drilling on the property do not exist and could therefore not be sampled by Tetra Tech EBA as of the Effective Date of the previous filing, March 11, 2012. Of the 868 historic drillholes used in the resource estimate, representing 70% of total drilling, 271 (31.2%) of these holes have available assay certificates. An additional 390 (44.9%) have available lithology/assay compilation reports available for digital data validation. In total, 661 historic drillholes (76.1%) have some form of supporting documentation available for validation. Table 12-1 summarizes the historical information that was available for review by Tetra Tech EBA.

Tetra Tech EBA applied geostatistical comparison in addition to review of the available supporting documentation to validate and support inclusion of historical assay data into the mineral resource estimates presented in this report. The review aimed to establish that the use of RC drillhole data is reasonable and is not positively biased and that recent drilling supports the grade reported in historic drilling. The verification has been conducted exclusively on the data being incorporated into the resource estimates included in this report, namely for the Big Rock, Fosters-Canadian, North Slope, Bohemian, Schooner, Sleeman and Classic mineralized zones.

Drillhole location and orientation data used in the database has been extracted and from the original Viceroy AutoCAD database, verified with available logs and survey reports and retranslated from historic mine grid co-ordinates to UTM co-ordinates. Historical Viceroy surveying was completed from 1996 onwards using survey grade Trimble equipment which co-measured Lat/Lon co-ordinates and the Viceroy mine grid. Control of the surface co-ordinates was completed by Golden Predator in 2009-2010 with the assistance from the original Viceroy surveyor. In total, approximately 40 historical drillhole collar monuments were located, mostly as stakes with labelled aluminum tags, between west Big Rock and Schooner and used to define an accurate transformation from the original mine grid to modern UTM co-ordinates that could be applied to all historical drillholes and surveyed information. The process was completed using an affine polynomial alogorithm and was verified using actual road centrelines and later with the 2011 LIDAR survey conducted on the property. In 2010, it was determined by Golden Predator that an upward 2.49 metre vertical shift be applied to the historical datum used by Viceroy. Comparison of the re-surveyed historical monuments to the transformed database co-ordinates results in location deviation ranging from approximately 0.5 to 2 metres at the lateral extents of the property. Tetra Tech EBA has reviewed the database and methodology used to undertake this transformation and feels that it has been completed using acceptable and modern methods.

Table 12-1: Summary of Drillhole Information Available for Review, Effective March 11, 2012

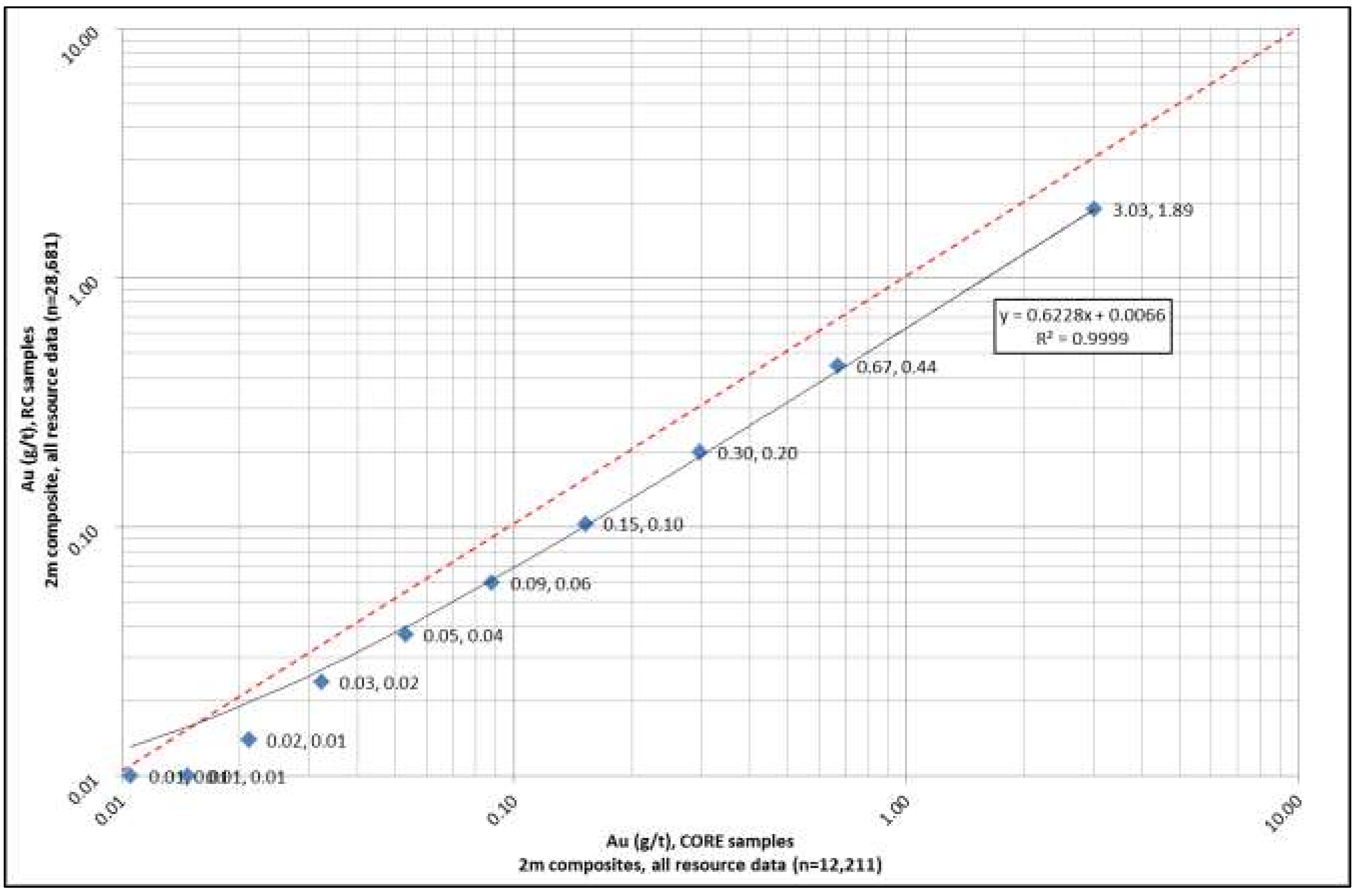
Drill Series	Year Drilled	Operator	Drill Type	# of Holes	Total Meters		Resource Holes (count)	% of total drill count	# DH's with Cert.'s	# DH's with Compilation's	Cert's. + Compilation's	%Assay Cert. (availability)	QaQc Data (availability)	Security Procedures
RC89	1989	Norex	RC	13	1,704	Historic	5	0.4%	0	5	5	0%	0%	?
DD89	1989	Norex	Core	9	1,097	Historic	7	0.6%	0	0	0	0%	0%	?
RC90	1990	Loki	RC	309	14,838	Historic	94	7.6%	92	2	94	98%	0%	?
DD90	1990	Loki	Core	16	1,090	Historic	8	0.6%	0	0	0	0%	0%	?
PQ90	1990	Loki	Core	5	198	Historic	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
RC91	1991	Loki	RC	348	18,007	Historic	67	5.4%	67	0	67	100%	0%	?
DD91	1991	Loki	Core	34	1,645	Historic	3	0.2%	3	0	3	100%	0%	?
RC92	1992	Loki	RC	19	1,236	Historic	8	0.6%	0	8	8	0%	0%	?
RC93	1993	Loki	RC	151	8,542	Historic	65	5.2%	0	56	56	0%	0%	?
RC94	1994	Loki	RC	242	10,891	Historic	132	10.6%	0	130	130	0%	0%	?
RC95	1995	Loki	RC	317	14,981	Historic	193	15.6%	0	189	189	0%	0%	?
DD95	1995	Loki	Core	25	1,200	Historic	12	1.0%	0	0	0	0%	0%	?
RC96	1996	Viceroy	RC	271	14,458	Historic	47	3.8%	47	0	47	100%	0%	?
DD96	1996	Viceroy	Core	23	2,992	Historic	5	0.4%	1	0	1	20%	0%	?
RC97	1997	Viceroy	RC	367	23,045	Historic	114	9.2%	52	0	52	46%	0%	?
RC98	1998	Viceroy	RC	219	13,960	Historic	87	7.0%	4	0	4	5%	0%	?
DD98	1998	Viceroy	Core	10	662	Historic	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
RC99	1999	Viceroy	RC	53	4,244	Historic	16	1.3%	0	0	0	0	0%	?
BC04	2004	Spectrum	Core	5	770	Historic	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
BC06	2006	Alexco	Core	9	1,171	Historic	5	0.4%	5	0	5	100%	100%	Documented
BC09	2009	Golden Predator	Core	30	4,981	Recent	22	1.8%	22	0	22	100%	100%	Documented
BC10	2010	Golden Predator	Core	13	2,414	Recent	8	0.6%	8	0	8	100%	100%	Documented
RC10	2010	Golden Predator	RC	16	2,352	Recent	4	0.3%	4	0	4	100%	100%	Documented
BC11	2011	Golden Predator	Core	209	31,210	Recent	199	16.0%	199	0	199	100%	100%	Documented
RC11	2011	Golden Predator	RC	135	24,300	Recent	125	10.1%	125	0	125	100%	100%	Documented
BCS	2011	Golden Predator	Sonic	18	266	Recent	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
BC12 (thru 401)	2012	Golden Predator	Core	18	1,995	Recent	14	1.1%	18	0	18	129%	100%	Documented
Total DH's in DB at time of EBA resource:				**2,884**			**1,240**							

12.1.1 Tetra Tech EBA Comparison Between Core and RC Data

Work in the 1980's and 1990's by Loki and Viceroy identified that drill core recoveries were low and that the use of RC drilling on the property could provide more reliable recovery of mineralization during exploration and delineation campaigns. Evidence in the database exists to support internal Viceroy twinning programs to monitor gold grade reporting. Tetra Tech EBA did not locate specific results for the twin programs and did not conduct a detailed analysis of any historical twin holes reported in the drillhole database. RMI compared Viceroy and Loki RC samples collected in the 1990's against Golden Predator diamond corehole samples (see Section 12.4). The older RC samples were spatially paired with the newer core hole samples and in general showed the RC samples to be lower grade than the diamond core samples.

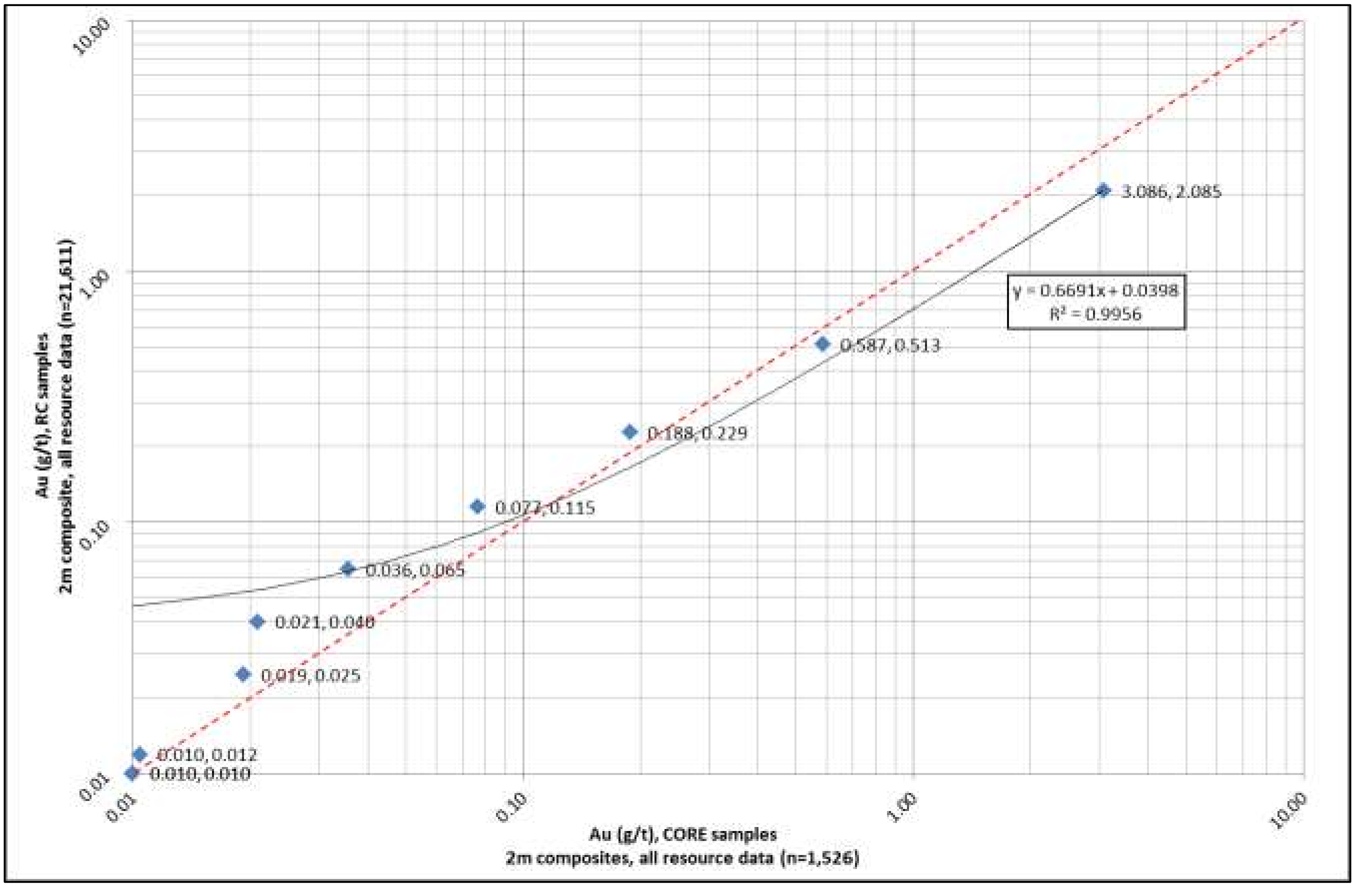
Using the 2 metre composite dataset, decile-decile plots were created and analyzed for apparent bias. The data set was filtered to remove the low grade composites below 0.01 g/t Au in order to reduce the impact of null and low range detectible gold grades. No upper grade caps were applied. The results of core sample and RC samples for both historical and recent drilling within the main mineralized areas is shown below in Figure 12–1. The plot identifies a slight bias in gold grades reporting higher for core samples than the RC samples. The effect of this is felt by Tetra Tech EBA to be insignificant given the scale of the bias and given the overall number of samples reported as RC (n=28,681) versus core (n=12,211).

Figure 12–1: Decile-decile Plot Comparing 2 Metre Composites of all Core and RC Gold Data Included in this Report



A second comparison included a refinement to the dataset by restricting the sample comparison to only those drilled by Loki/Viceroy drilling. The results of this comparison are seen in Figure 12–2. Low grade ranges below 0.1 g/t Au appear to bias slightly towards the RC sampling, where grades plotting above 0.2 g/t Au plot near the unity line. Again, it is noted that the number of historical RC samples (n=21,611) far exceeds that of the core samples (n=1,526).

Figure 12–2: Decile-decile Plot Comparing 2 Metre Composites of Historical Viceroy/Loki Core and RC Gold Data Included in this Report



12.1.2 Tetra Tech EBA Comparison Between Recent Core and Historical Core

Recent drilling conducted by Golden Predator has aimed to test the validity of historically reported gold grades. The traces of 12 historical holes lie within 7 metres of Golden Predator holes (Table 12-2). The recent twin pairs were visually inspected using geological software and found to compare favorably in terms of mineralization depth, intercept thickness, grade and logged lithology. The majority of Golden Predator drilling lies with 25 metres of Loki-Viceroy era holes. In general, nearby holes from the historical drill dataset show strong similarities in the intercept thickness, tenor and logged lithologies with Golden Predator drilling.

Table 12-2: Recent Twins of Historical Drillholes

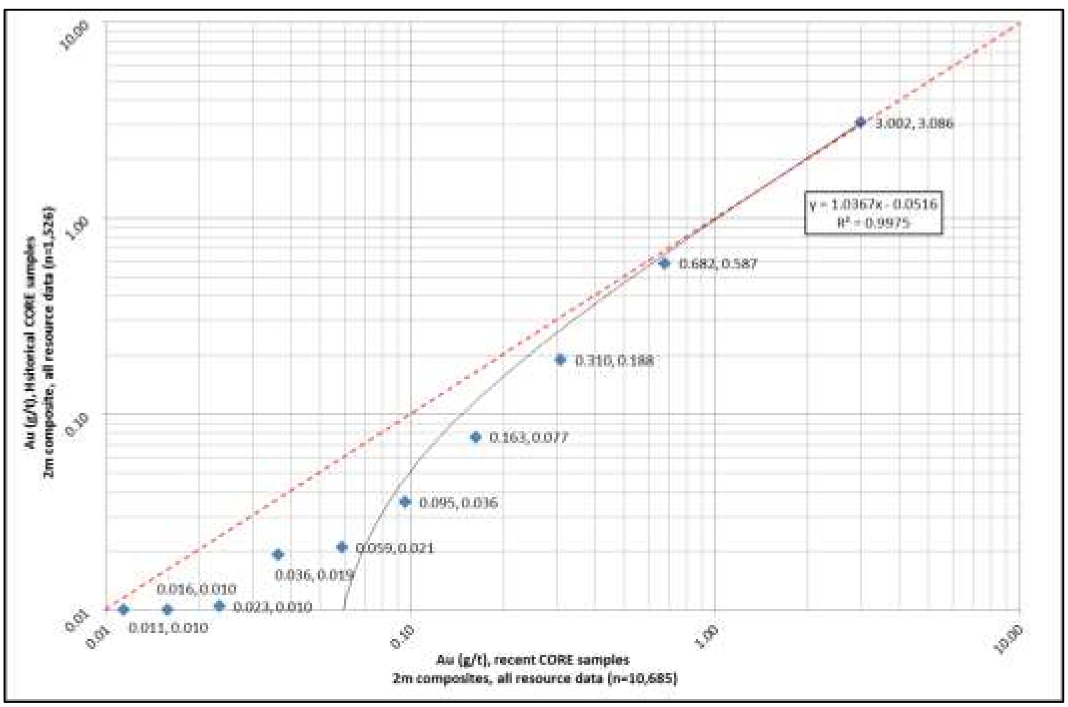
Historic Resource Hole	Offset GPY Hole	Area
RC97-1967	BC11-236	Bohemian
RC98-2145	BC11-189	Bohemian

Table 12-2: Recent Twins of Historical Drillholes

Historic Resource Hole	Offset GPY Hole	Area
RC95-1363	BC11-357	East Big Rock
RC96-1570	RC11-2433	East Big Rock
RC96-1623	RC11-2432	East Big Rock
RC97-1902	BC11-358	East Big Rock
RC97-1772	RC11-2409	North Slope
RC97-1773	BC11-300	North Slope
RC98-2198	BC11-196	Schooner
RC99-2267	BC10-210	Schooner
RC96-1577	RC11-2458	West Big Rock

Statistical comparison was made between the historical core gold grade values versus the recent core gold grade values reported by Golden Predator (up to and including core hole BC12-401). Figure 12–3 shows a decile-decile comparison of the two datasets to reveal that at grades generally below 0.2 g/t Au recent drilling plots higher than historical drilling and at ranges greater than 0.2 g.t Au (ie. > 75th percentile) that historical gold grades plot near to unity with the recent drilling. Tetra Tech EBA feels this is reasonable support for the sampling trend given that much of the recent drilling has been targeting known areas of mineralization and generally contains less lower grade material as would have been recovered in historical regional exploration and geotechnical core drilling programs.

Figure 12–3: Decile-decile Plot Comparing 2 Metre Composites of Recent Golden Predator Core and Historic Core Gold Data Included in this Report



12.1.3 Tetra Tech EBA Statement

Tetra Tech EBA feels that the historical drilling data is verifiable and valid for use in Mineral Resource estimation. Support is based on the review of historical results, positive comparison of the historical results to recent Golden Predator drilling and minimal to no bias apparent between the various datasets.

12.2 Tetra Tech EBA Verification of Recent Data

A site visit was conducted between March 19-21, 2012, by Tetra Tech EBA geologist and Independent Qualified Person (QP) James Barr, P.Geo. The purpose of the visit was to become familiar with the site layout and facilities, review core logging and sample handling procedures, review drill core and collect core samples from recent Golden Predator drilling for independent analysis. Mr. Barr was accompanied by Golden Predator Senior Geologist Bruce Otto, Geologist Mark Shutty and Program Manager Don Penner for the duration of the visit. A second site visit was conducted from May 30-31, 2012, at which time no QA/QC review or sample collection was completed.

In total, 7 core holes were reviewed while on site in March 2012, which provided a familiarity of the variety of rock types and mineralizing systems present at Brewery Creek. Specific core intervals from these holes were selected based on availability, spatial distribution and representative grades. During this field visit 6 samples were collected from 4 holes, packaged in sampling bags, and transported by Mr. Barr to the Tetra Tech EBA offices in Vancouver and then couriered directly to ALS Chemex laboratories for analysis.

For QA/QC purposes a Standard Reference Material (SRM) and a blank sample was included in the sample batch for a total of 8 samples for laboratory analysis at ALS Chemex (Vancouver). Table 12-3 presents the results of the ALS Chemex tests, labelled as Tetra Tech EBA, against the original Golden Predator analytical values for Au, and Ag.

Table 12-3: Independent Drill Core Samples Collected by Tetra Tech EBA

<table>
<tr><th>Hole Id</th><th>From</th><th>To</th><th>Company</th><th>Sample</th><th>Rockcode *</th><th>SG</th><th>Au (g/t)</th><th>Ag(g/t)</th><th>Au g/t RS **</th></tr>
<tr><td rowspan="3">BC11-360</td><td rowspan="3">80</td><td rowspan="3">82</td><td>Golden Predator</td><td>1294244</td><td rowspan="3">SY/IS</td><td rowspan="3">2.68</td><td>0.77</td><td>-</td><td>-</td></tr>
<tr><td>Tetra Tech EBA</td><td>500408</td><td>1.07</td><td>0.9</td><td>1.13</td></tr>
<tr><td colspan="2">% Difference</td><td>33.1</td><td>-</td><td>37.89</td></tr>
<tr><td rowspan="3">BC11-333</td><td rowspan="3">28.73</td><td rowspan="3">30.35</td><td>Golden Predator</td><td>1327702</td><td rowspan="3">LAQM/IQM</td><td rowspan="3">2.55</td><td>7.85</td><td>-</td><td>-</td></tr>
<tr><td>Tetra Tech EBA</td><td>500409</td><td>11.15</td><td>0.3</td><td>11.65</td></tr>
<tr><td colspan="2">% Difference</td><td>34.7</td><td>-</td><td>38.97</td></tr>
<tr><td rowspan="3">BC11-333</td><td rowspan="3">52.9</td><td rowspan="3">54.25</td><td>Golden Predator</td><td>1327718</td><td rowspan="3">SGW/SNG</td><td rowspan="3">2.67</td><td>14.60</td><td>-</td><td>-</td></tr>
<tr><td>Tetra Tech EBA</td><td>500410</td><td>16.05</td><td>0.9</td><td>16.6</td></tr>
<tr><td colspan="2">% Difference</td><td>9.46</td><td>-</td><td>12.82</td></tr>
<tr><td rowspan="3">SRM</td><td rowspan="3"></td><td rowspan="3"></td><td>Tetra Tech EBA</td><td>500411</td><td rowspan="3"></td><td rowspan="3">n/a</td><td>13.45</td><td>4.10</td><td>-</td></tr>
<tr><td>CDN-GS-13A</td><td>n/a</td><td>13.20±0.72</td><td>-</td><td>-</td></tr>
<tr><td colspan="2">% Difference</td><td>1.88</td><td>-</td><td>-</td></tr>
<tr><td rowspan="3">BC11-293</td><td rowspan="3">60</td><td rowspan="3">62</td><td>Golden Predator</td><td>K739669</td><td rowspan="3">LAQM/IQM</td><td rowspan="3">2.57</td><td>7.64</td><td>2.50</td><td>-</td></tr>
<tr><td>Tetra Tech EBA</td><td>500412</td><td>9.72</td><td>3.00</td><td>10.05</td></tr>
<tr><td colspan="2">% Difference</td><td>24.0</td><td>18.2</td><td>27.25</td></tr>
<tr><td rowspan="3">BC11-321</td><td rowspan="3">71.2</td><td rowspan="3">72.7</td><td>Golden Predator</td><td>1292722</td><td rowspan="3">AQM/IQM</td><td rowspan="3">2.63</td><td>20.60</td><td>13.00</td><td>-</td></tr>
<tr><td>Tetra Tech EBA</td><td>500413</td><td>5.91</td><td>18.60</td><td>5.99</td></tr>
<tr><td colspan="2">% Difference</td><td>110.8</td><td>35.4</td><td>109.89</td></tr>
<tr><td>Blank</td><td></td><td></td><td></td><td>500414</td><td></td><td>2.77</td><td>0.03</td><td><0.2</td><td>0.03</td></tr>
</table>

Table 12-3: Independent Drill Core Samples Collected by Tetra Tech EBA

Hole Id	From	To	Company	Sample	Rockcode *	SG	Au (g/t)	Ag(g/t)	Au g/t RS **
BC11-321	74.2	75.7	Golden Predator	1292725	AQM/IQM	2.66	4.78	14.00	-
			Tetra Tech EBA	500415			3.44	5.00	3.51
			% Difference				*32.6*	*94.7*	*30.64*

* Client rock code/Tetra Tech EBA rock code

** ALS Chemex re-sample value

The samples were analysed using the following ALS Chemex laboratory methods:

- Prep 31 (Split off 250g and pulverize split to better than 85% passing 75 microns),
- Specific Gravity – OA-GRA08A
- Ore Grade 30g nominal sample weight– Au-AA25
- Analytes & Ranges – ME-ICP41

Tetra Tech EBA conducted a percent difference comparison of the original Golden Predator values against the analytical results provided by ALS Chemex. A percent difference is used to provide an absolute difference between the duplicate samples relative to their mean allowing meaningful comparison independent of the magnitude of the individual grades. The analysis was calculated using the following formula where, Golden Predator is the original analytical result, and Tetra Tech EBA is the duplicate analytical result obtained from ALS Chemex.

Equation 1: Percent difference comparison

$$\%\,Difference = \left| \frac{(GPD - EBA)}{\frac{(GPD + EBA)}{2}} \right| \times 100\%$$

12.2.1 Tetra Tech EBA Discussion of Independent Sample Results

Through discussion and observations made while on site, Tetra Tech EBA confirms that Golden Predator is using best practices in their exploration and sample collection procedures.

Results from the independent sample collection using percent difference analysis show that in 4 of the 6 samples tested, the Golden Predator samples graded lower (Au g/t) than that of the Tetra Tech EBA samples (ALS Chemex) analysis. Golden Predator samples 1292725, and 1292722 were exception to this with +110.8% and +32.6% differences, respectfully.

Due to the irregularities found in the percent difference comparison for Tetra Tech EBA sample 500413, sample re-analysis was requested at ALS Chemex. The results for the re-sampling indicate slight global increase in all reported gold grades. The results, however, do support consistent values and reproducibility of the grades as seen in Table 12-3.

Specific gravity (SG) for each sample was tested and fall within the ranges of values determined by Golden Predator work. This analysis showed no major deviation in the results in terms of the tested lithologies and analytical results.

12.2.2 Tetra Tech EBA Statement on Recent Data Verification

Tetra Tech EBA sampling conducted on site indicated a slight variance in grade results for all samples collected on site. The positive percent difference found in hole BC11-321 was exceptionally high and may be accountable to a core recovery issue following sampling or to material shifting within the core box as the material was broken and integrity was quite poor. A number of factors could account for this deviation; however, it is not felt that a bias is present in the dataset.

Based on the visual inspection of core, review of sampling methodology and independent sampling, Tetra Tech EBA feels that the results reported by recent Golden Predator drilling is reliable and that inclusion of this data for mineral resource estimation is supported.

12.3 RMI Data Verification

A site visit was conducted between October 16 and October 18 2012, by RMI geologist and Independent Qualified Person (QP) Michael Lechner, P.Geo. The purpose of the visit was to become familiar with the site layout and facilities, review core drilling procedures, review core logging/sample handling procedures, examine drill core and review electronic data collection practices. Mr. Lechner was accompanied by Golden Predator Senior Geologist Bruce Otto and Project Geologist Tyler Bourne.

While on site, Mr. Lechner examined two diamond drill rigs that were operating in the Classic-Lone Star areas. The first drill rig that was visited was an A5 drill operated by Matrix Diamond Drilling Inc. (drillhole BC12-580). The hole was approximately 250 metres deep at the time of the visit. The drill site was clean and the core was correctly handled at the site. The second drill rig that was visited was operated by Kluane Drilling Ltd. (drillhole BC12-576). Both drill rigs appeared to be delivering nearly 100% recovery. Both drills were using NQ tools with 10-foot-long core barrels.

Portions of three recent and two older diamond drillholes were examined by Mr. Lechner while on site. Selected drillhole intervals from specified holes were compared against the original drill logs. Table 12-4 lists the holes and intervals that Mr. Lechner examined.

Table 12-4: Drill Core Samples Examined by RMI

Drillhole	Area	Depth (m)	Comments
BC12-438	West Big Rock	41.15 to 76.50	Black argillaceous sediments and LAQM
BC12-440	West Big Rock	38.94 to 65.95	Intersection of +2 g/t LAQM (46.35 - 57.45)
BC12-451	West Big Rock	29.20 to 60.25	Examples of low and high grade LAQM mineralization
DD95-0061	Fosters	2.60 to 25.70	Highly weathered/altered carbonaceous sediments
DD95-0062	Fosters	6.10 to 26.65	Highly altered well mineralized LAQM

RMI verified 10% of the 2012 drillhole assays (from drillhole BC12-411 onward) by comparing electronic database gold assay records against signed assay certificates. Table 12-5 summarizes the drillholes that were audited by RMI.

Table 12-5: Drillhole Assay Samples Verified by RMI

Area	Drillhole	No. Assays	No. Meters
Bohemian	BC12-418	48	95.10

Table 12-5: Drillhole Assay Samples Verified by RMI

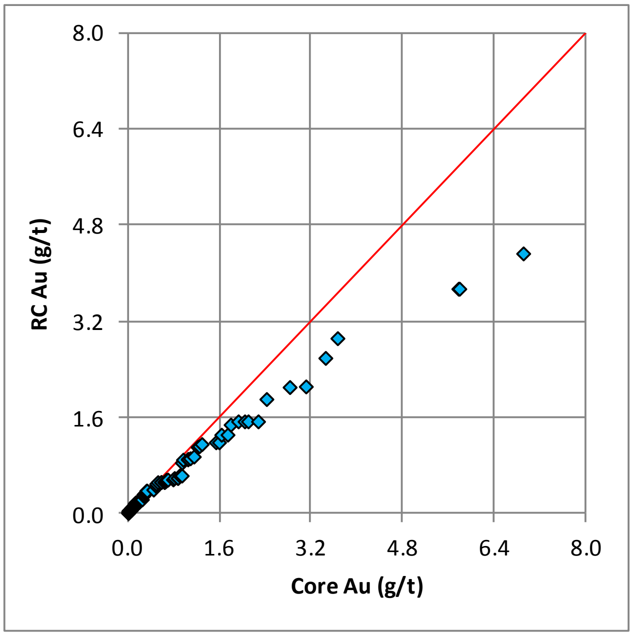
Area	Drillhole	No. Assays	No. Meters
Bohemian	BC12-423	60	112.77
Schooner	BC12-559	72	135.63
Schooner	RC12-2498	40	60.96
Fosters	RC12-2466	35	53.34
Fosters	RC12-2471	29	44.20
West Big Rock	BC12-411	65	120.39
West Big Rock	BC12-477	64	125.58
West Big Rock	BC12-478	78	142.32
East Big Rock	BC12-483	48	94.80
East Big Rock	BC12-546	77	137.16
Classic	RC12-2500	113	172.21
Classic	RC12-2513	197	300.23
Lone Star	BC12-580	185	340.46
Lone Star	RC12-2523	123	187.45
Grand Total	N/A	1,234	2,122.60

12.3.1 RMI Comparison of Core Versus RC Drilling

RMI spatially paired 6-metre-long diamond core composites with 6-metre-long reverse circulation (RC) samples for the Bohemian, Schooner, Fosters, West Big Rock, East Big Rock, and Classic-Lone Star deposits. The data were paired by lithology and oxidation constraints (i.e. oxidized intrusive material only). A similar pattern was observed for all but the Classic-Lone Star data where the diamond drilling data tended to be slightly higher grade than nearby RC samples. This relationship was found to be reversed for the Classic Zone, which showed the RC samples to be higher grade than nearby diamond core samples.

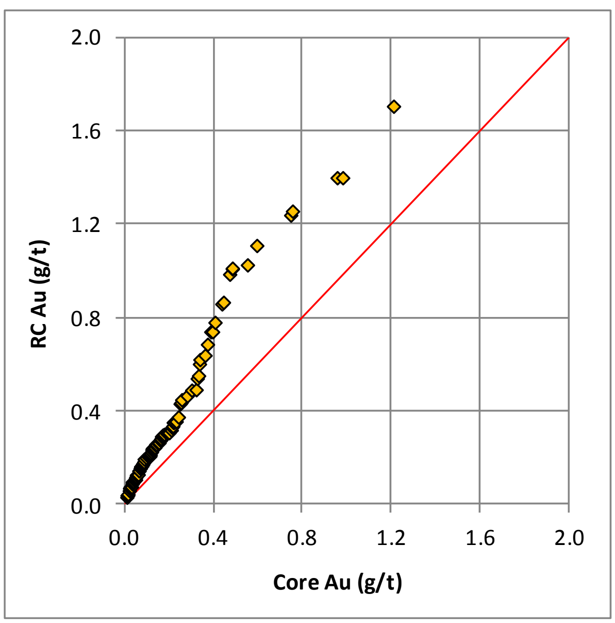
Figure 12–4 is a quantile-quantile (QQ) plot that compares diamond core gold grades against RC gold grade samples. A maximum sample separation distance of 25 metres was used and both sample types represent oxidized intrusive material.

Figure 12–4: QQ Plot Comparing Diamond Core and RC Samples - Bohemian Deposit



The data in Figure 12–4 show that there is an apparent high bias associated with the diamond drilling sample data relative to nearby RC data. As mentioned above, this relationship was also seen with respect to the Schooner, Fosters, West and East Big Rock deposits. The opposite relationship was observed for the Classic deposit as depicted by the QQ plot shown in Figure 12–5.

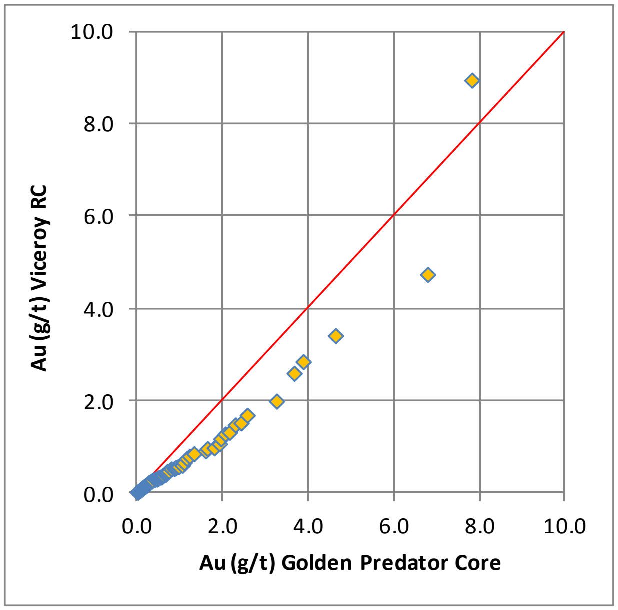
Figure 12–5: QQ Plot Comparing Diamond Core and RC Samples - Classic Deposit



More specific gold grade comparisons were made by spatially pairing 6m drillhole composites (intrusive material only) that were collected by different companies using different methods. A maximum separation distance of

50 metres was used in pairing the two data types. Figure 12–6 is a QQ plot that compares Golden Predator core samples (X-axis) against Viceroy RC samples (Y-axis) for the Bohemian deposit.

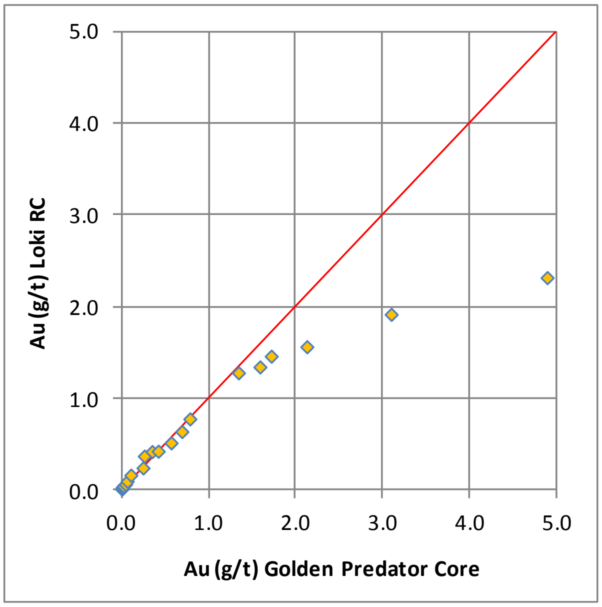
Figure 12–6: QQ Plot Comparing Golden Predator Diamond Core and Viceroy RC Samples - Bohemian Deposit



The data in Figure 12–6 show that there is a low bias associated with the older Viceroy RC samples when that data is compared against the more recent Golden Predator core samples.

Figure 12–7 is a QQ plot that compares Golden Predator core samples (X-axis) against Loki RC samples (Y-axis) for the lower Fosters deposit.

Figure 12–7: QQ Plot Comparing Golden Predator Diamond Core and Loki RC Samples - Fosters Deposit



There is a reasonably close comparison between the older Loki RC samples (Y-axis) and the newer Golden Predator core samples (X-axis) for gold grades below 1.5 g/t. above approximately a 1.5 g/t cut-off grade, the older data is biased low.

Figure 12–8 and Figure 12–9 compare older Viceroy RC gold samples against newer Golden Predator core and RC samples, respectively for the East Big Rock deposit.

Figure 12–8: QQ Plot Comparing Golden Predator Diamond Core and Viceroy RC Samples - East Big Rock Deposit

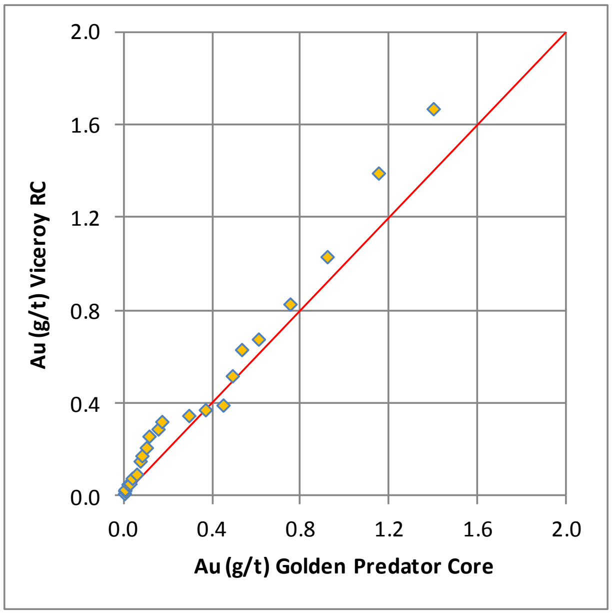
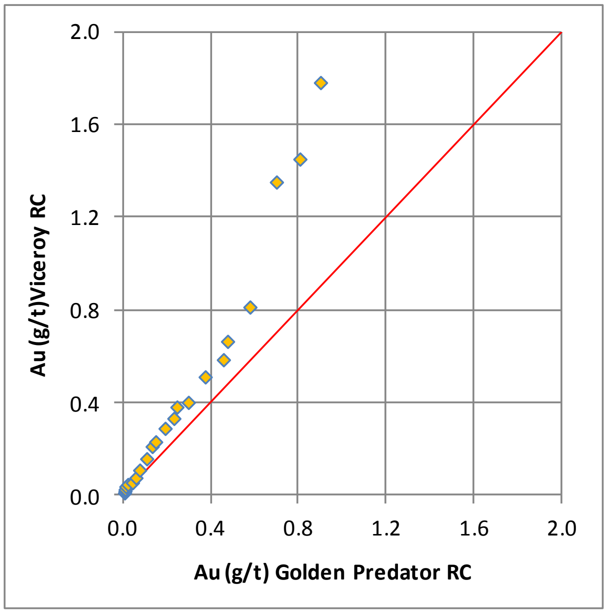


Figure 12–9: QQ Plot Comparing Golden Predator RC and Viceroy RC Samples - East Big Rock Deposit

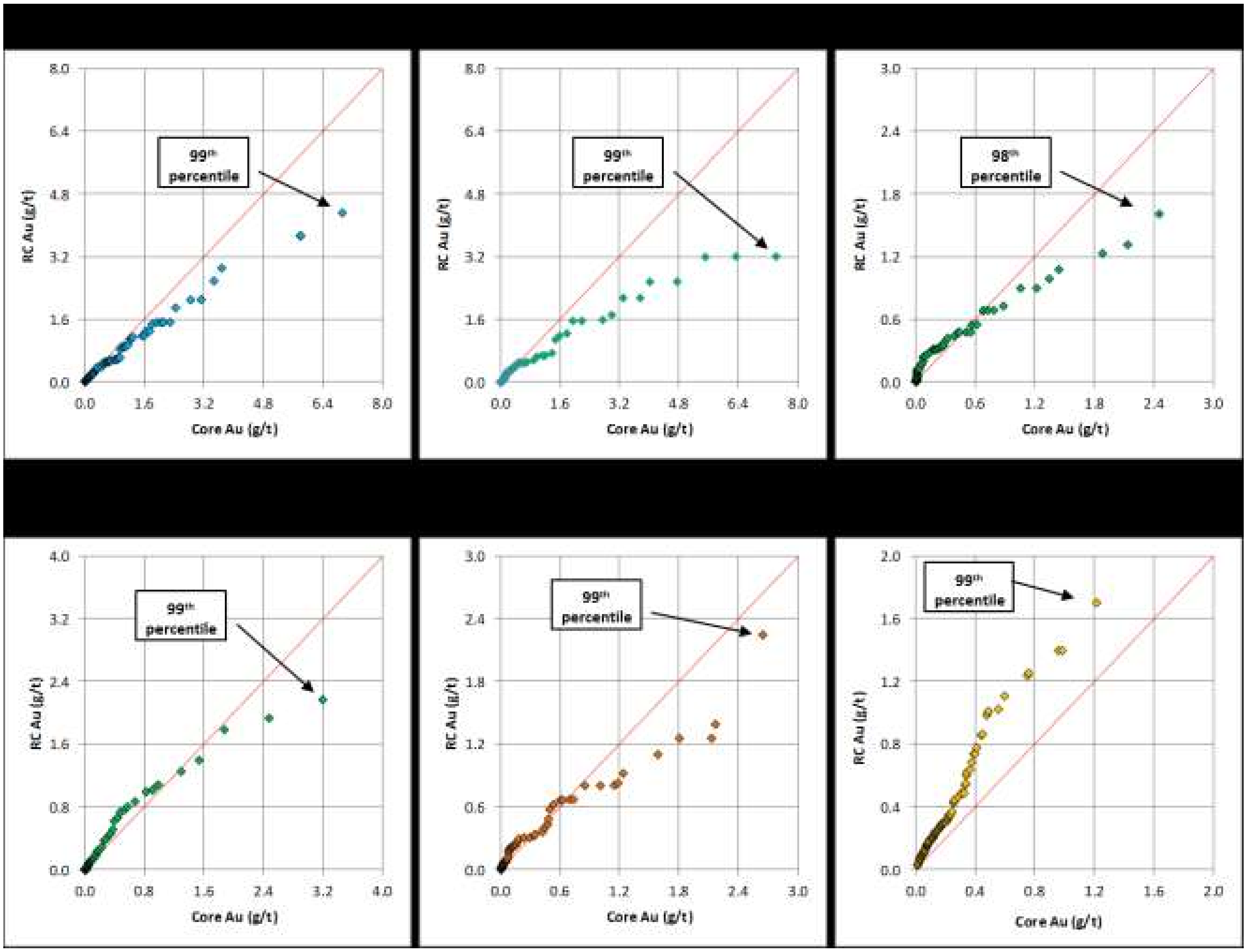


The QQ plot data Figure 12–8 and Figure 12–9 that compare Viceroy RC data against Golden Predator core and RC samples show that there is a slight high bias associated with the Viceroy RC data.

12.3.2 Drilling-Sampling-Recovery Factors

RMI compared diamond core sample data with nearby reverse circulation (RC) samples to see if there were any significant differences in gold grades. The original assay samples were composited to 6 m lengths and then core and RC samples were spatially paired provided both samples types were collected from oxidized intrusive material. RMI notes that there is a slight to moderate high-grade bias associated with core hole samples collected from the Bohemian, Schooner, Fosters, West Big Rock, and East Big Rock deposits. The opposite relationship (i.e. RC samples were higher than core) was observed with the Classic-Lone Star deposit data. At this stage of exploration at Classic it is difficult to determine the cause behind these apparent differences. Groundwater is often the cause for poor RC sampling results but according to Golden Predator's geologic staff, groundwater should not be an issue with the RC samples at Classic. Figure 12–10 contains six quantile-quantile (QQ) plots that compare RC gold grades (Y-axis) with core gold grades (X-axis). As mentioned above, corehole assays from most of the deposits tend to be higher than the RC data above a 0.5 to 1.0 g/t cut-off grade.

Figure 12–10: Core vs. RC Samples



RMI compared gold grades with core recovery for the Classic-Lone Star deposit to see if gold was being lost when recovery was poor. Gold grades are seen to increase marginally as core recovery increases. It is possible that the core samples are not as representative as RC samples at Classic but RMI is recommending that Golden Predator drill three to five diamond holes next to existing RC holes to further examine biases.

Based on the sample studies that have been completed it is RMI's opinion that the core and RC samples are suitable to be used to estimate Mineral Resources for the Bohemian, Schooner, Fosters, West Big Rock and East Big Rock zones. Because of wider spaced drilling and the potential for biased RC samples, RMI elected to classify Classic and Lone Star as Inferred Resources.

12.3.3 RMI Statement Regarding Data Verification

RMI examined sampling and assaying procedures that were implemented by Golden Predator and also verified that assay records from the 2012 drilling campaign were accurately entered into the project database. Various diamond core and RC sample data were spatially paired and then compared with one another to check for possible biases. In general, most of the spatially paired sample comparisons suggest that the older RC sample data are biased low when compared against Golden Predator core data.

Based on RMI's verification procedures the gold assay data used for the Brewery Creek Property are suitable for estimating Mineral Resources. RMI recommends that Golden Predator follow up on the apparent high-bias associated with Classic and East Big Rock RC samples by drilling two or three core holes adjacent to existing RC holes so that possible biases can be further analyzed.

12.4 Data Verification by Gustavson

12.4.1 Verification of Historical Data

To validate historical drilling, Gustavson performed a point validation analysis on the historical data. Historical data were used to estimate at gold grade value at the XYZ location of Golden Predator samples using an Inverse Distance Squared method. The estimated values were then compared to the actual value of the Golden Predator sample at that location. A correlation analysis was then performed on the estimated versus actual data. Point validation was restricted by major rock type (intrusive or sedimentary). The results of the correlation analyses were mixed, and Gustavson performed other analyses to confirm historical data. Gustavson next compared the historical drillhole and blasthole data on a bench by bench basis. This analysis showed that the two independent assay campaigns showed similar high grade zones in blastholes and nearby drillholes. Visual inspection showed a good correlation, though it indicated there might be down-hole drift in the drillholes. Gustavson also visually compared historical and Golden Predator drillhole grades on a section basis. Again, high grade zones indicated in the historical data were matched by high grade assays in modern drilling. This visual inspection also showed that some of the previously noted discrepancies between historical and modern drilling may be due to the location of drilling campaigns. Historical campaigns generally targeted the core of the high grade portions of the deposits, while modern drilling has focused more on the periphery of the deposits. Modern drilling has also taken place after mining operations were begun, and therefore is affected by the removal of material that was present during past campaigns. There were no twin holes of historical and Golden Predator drilling to compare. Given the analyses noted above, Gustavson is of the opinion that the historical data is appropriate for resource estimation.

12.4.2 Verification of Drill Data

Drillhole collar data were compared to the site topographic map to confirm that elevations are consistent. The survey data for each drillhole were also examined. Those drillholes containing greater than 5° variation in dip or azimuth, or containing greater than 1° per metre are verified by Golden Predator, and corrected, if necessary. To ensure logging quality, Golden Predator verifies assay and lithology data entries to ensure that data are available from top to bottom of drillhole, with no missing intervals or intervals exceeding the total depth of drillhole. Gustavson reviewed blasthole data compared to the site pre and post mining topography to check that elevations were consistent with those data.

Gustavson implements a data validation step to ensure that the Excel database received matches the actual lab assay certificates. Gustavson requires a 99% confidence level and minimum 4% confidence interval to consider a database of good enough quality for resource estimation. The Golden Predator sample assay Excel database contains 3548 samples. Of the 502 samples versus assay certificate values checked, Gustavson found 45 errors. However, it was noted that in some cases the assay values had been rounded from three decimal places to two, which may account for some errors. This validation produced a 3% confidence interval at a 99% confidence level, which Gustavson believes is an acceptable error rate and that the data is valid.

12.4.3 Gustavson Site Visit

Gustavson QP, M.C. Newton, III, visited the site on June 4 and 5, 2013. During the site visit, Dr. Newton examined rocks in the Lucky, Golden, Kokanee and Pacific pits, took structural measurements, and collected three independent grab samples for gold assay. During the course of the visit, the samples remained in Dr. Newton's custody or vision and were delivered personally by Dr. Newton to ALS Chemex in Whitehorse, Canada. The samples were assayed by ALS's ICP21 and gravimetric methods. Results of these three samples are provided in Table 12-6 below. BC-1 and BC-2 were quartz-pyrite veined quartz monzonite and sample BC-3 was highly fractured but weakly altered felsic intrusive rock. The results of the Gustavson sampling independently verify Golden Predator's drilling results.

Table 12-6: Independent Sample Results

Sample	Au	Latitude	Longitude	Elevation
Description	ppm	WGS84	WGS84	ft
BC-1, Lucky	1.175	64.05909	-138.18966	3,024.222
BC-2, Golden	3.99	64.0656	-138.17179	2,726.967
BC-3, Kokanee	0.044	64.05765	-138.21242	3,192.956

During the course of the site visit, drill sites were examined, photographed and located by GPS. Locations matched coordinates in the database. At the drill sites examined by Gustavson, a concrete slab had been placed around pipe or rebar protruding from the drillhole and metal markers recorded hole number, azimuth and inclination of the hole.

Diamond core and RC cuttings from several holes, collected in multiple programs by several different mining and drilling companies, were examined. Core sample intervals were generally determined by natural geologic breaks, with no interval being larger than 1.5 metres. RC cuttings were sampled at 1.5 metre intervals. Core is stored in wooden boxes labeled with drillhole number and metreage. Sample intervals in core boxes are marked with stapled paper tags or metal tags. Mineralized intervals in core and RC cuttings were examined and correspond well with the Golden Predator assay database. Core and RC cuttings are stored in covered buildings or boxes and are relatively safe from weather and secure as the road to the storage area is gated and locked.

12.4.4 Gustavson's Statement on Data Adequacy

It is the QP's opinion that the data presented are adequate for the purposes of this report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Precious metals will be recovered from low grade mineralized material by heap leaching. To prepare the ore for leaching, a crushing plant is planned. Crushed ore will be placed on the heap by truck and leached with a dilute cyanide solution. Precious metals will be recovered from the leach solution by ADR plant. A more complete process description is given in the following section 17.

Considerable historical metallurgical testing of material from the Brewery Creek Property has been conducted with early reports completed in 1988 by Loki Gold Corporation and test work by Kappes, Cassiday & Associates and Lakefield in the 1990s.

The following review summarizes procedures used to conduct sample preparation, test work and the metallurgical data developed from the column leach tests at McClelland. SGS conducted the gold extraction projections using a metallurgical software program, METSIM, for the column leach test data.

The following three sources of metallurgical information have been used to complete this PEA:

- Viceroy operated the Brewery Creek heap leach facility from 1997 through 2002 and operations reports from this period have been used in estimation of cyanide consumption.

- Metallurgical data have been extracted from a McClelland draft report, McClelland Job No. 3618, to estimate metallurgical performance of residue material from the existing heap leach pad. This study was conducted on sonic drill samples collected from the existing heap material and delivered to McClelland in October 2011.
- In 2012, Golden Predator delivered drill core interval samples obtained from seven deposits located within the Brewery Creek project to McClelland for a metallurgical study. The metallurgical study, Job Number MLI 3719, was completed and a final report issued in July 2013, and included sample characterization, bottle roll testing and column leach testing.

13.1 MLI Job No. 3719 Metallurgical Test Work

Thirty-two PQ diameter drill core composite samples from seven pits were subjected to head assay characterization, bottle roll testing and column leach testing. Samples for testing included areas of waste and low recovery as part of a variability study. Golden Predator described each composite in detail, including rock type, and gave reasons for composite selection. Rock types for this study are summarized as follows:

- LAQM - Limonitic Altered Quartz Monzonite
- AQM - Altered Quartz Monzonite
- ARG - Graphitic Argillite
- Syenite

13.2 Head Assay Analyses

The following Table 13-1 summarizes gold head assays conducted on the composite samples. This data was extracted from the bottle roll testing results provided by McClelland on 21 December 2012.

Table 13-1: Drill Core Composites Gold Head Assay Results

Metallurgical Tests			Au Head Assay (g/t)	NaCN Sol. Au (%)
Ore Zone	Rock Type/Interval	Composite I.D.		
West Big Rock	LAQM/23-35m	BC12-01	2.20	94
	LAQM/70-82m	BC12-02	0.90	61
	AQM/50-60m	BC12-03	1.38	87
	ARG/51-59m	BC12-04	0.99	3
	LAQM/41-52m	BC12-27	1.60	94
East Big Rock	LAQM+ARG/28-41m	BC12-05	1.25	57
	LAQM/5-15m	BC12-06	0.92	72
	LAQM/30-40m	BC12-07	0.39	64
	LAQM/66-75m	BC12-08	1.07	53
Lower Fosters	LAQM/2-12m	BC12-09	0.26	65
	LAQM/16-30m	BC12-10	0.59	83
	LAQM/15-28m	BC12-11	0.60	77
	LAQM+ARG/2-14m	BC12-12	1.95	86
	AQM/33-40m	BC12-13	4.03	12

Table 13-1: Drill Core Composites Gold Head Assay Results

Metallurgical Tests			Au Head Assay (g/t)	NaCN Sol. Au (%)
Ore Zone	**Rock Type/Interval**	**Composite I.D.**		
Bohemian	LAQM/18-30m	BC12-14	0.29	66
	LAQM/30-42m	BC12-15	0.35	80
	LAQM/2-14m	BC12-16	0.63	110
	LAQM/12-19m	BC12-17	0.34	103
Schooner	LAQM/7-19	BC12-18	0.45	80
	LAQM/19-31m	BC12-19	5.10	86
	LAQM/31-43m	BC12-20	3.10	96
	LAQM/34-53	BC12-28	6.47	45
Moosehead	AQM/68-79m	BC12-21	1.29	4
	LAQM/12-25m	BC12-22	0.69	42
	LAQM/25-37mm	BC12-23	0.44	9
	AQM/37-49m	BC12-24	1.17	17
	AQM/63-78m	BC12-25	0.51	6
	AQM+ARG/78-91m	BC12-26	1.61	2
Classic	Syenite/3-17m	BC12-29	0.29	93
	Syenite /113-129m	BC12-30	0.60	68
	Syenite/47-62m	BC12-31	0.61	103
	Syenite/152-170m	BC12-32	0.39	80

The gold grade of the composite samples varied from 0.26 to 6.27 grams per tonne. The sodium cyanide soluble gold assays varied from completely refractory, near zero percent soluble, to completely amenable, 100 percent soluble.

13.2.1 Screen Analyses

Each composite sample was screened and stage crushed to approximately 80 percent passing 9.5 mm prior to bottle roll and column leach testing. These screen analysis are shown in the following Table 13-2.

Table 13-2: Drill Core Composites Head Screen Analyses

Ore Zone	Composite ID	Column ID	Weight Passing (%)					
			9.5 mm	**6.3 mm**	**1.7 mm**	**420 µm**	**150 µm**	**75 µm**
West Big Rock	BC12-01	P-1	81	56	26	13	8	6
	BC12-02	P-2	80	61	28	14	9	7
	BC12-03	P-3	79	54	23	11	6	4
	BC12-04	P-4	78	55	25	12	8	6
	BC12-27	P-27	78	49	20	9	6	5
East Big Rock	BC12-05	P-5	87	68	36	19	12	8
	BC12-06	P-6	85	64	33	17	11	7
	BC12-07	P-7	82	60	30	16	11	8
	BC12-08	P-8	80	47	19	8	5	4
	BC12-09	P-9	78	64	43	28	22	19

Table 13-2: Drill Core Composites Head Screen Analyses

Ore Zone	Composite ID	Column ID	Weight Passing (%)					
			9.5 mm	6.3 mm	1.7 mm	420 µm	150 µm	75 µm
Lower Fosters	BC12-10	P-10	83	62	30	14	8	6
	BC12-11	P-11	78	57	28	15	9	7
	BC12-12	P-12	87	77	46	23	14	10
	BC12-13	P-13	78	50	20	9	5	4
Bohemian	BC12-14	P-14	84	55	23	10	6	4
	BC12-15	P-15	79	45	16	7	4	3
	BC12-16	P-16	85	60	28	13	8	6
	BC12-17	P-17	84	57	22	10	6	5
Schooner	BC12-18	P-18	78	52	23	11	7	5
	BC12-19	P-19	77	51	19	8	5	4
	BC12-20	P-20	84	59	24	11	7	5
	BC12-28	P-28	80	52	22	9	5	4
Moosehead	BC12-21	P-21	77	50	21	10	6	5
	BC12-22	P-22	79	51	21	10	6	5
	BC12-23	P-23	77	45	15	6	4	3
	BC12-24	P-24	75	43	15	6	4	3
	BC12-25	P-25	79	52	19	9	5	4
	BC12-26	P-26	80	52	21	10	6	5
Classic	BC12-29	P-29	81	63	32	12	5	3
	BC12-30	P-30	78	51	24	11	6	4
	BC12-31	P-31	84	61	33	17	9	5
	BC12-32	P-32	82	57	29	16	9	6

Two samples, both from Lower Fosters, showed greater than 10 percent passing 75 microns. In the column test program, only these samples were agglomerated with cement.

13.2.2 Bottle Roll Testing

Bottle roll tests were conducted on 80 percent passing 9.5 mm composite test charges from each drillhole to determine gold extraction, gold leach kinetics and reagent consumption. The tests were conducted for 96 hours maintaining 40 percent solid, pulp pH between 10.8 and 11.2, sodium cyanide concentration of 1 g/l and pregnant leach solution samples were withdrawn at 2, 6, 24, 48, 72 and 96 hours to measure pH, cyanide concentration, gold and silver concentrations. At the end of 96 hours, the bottle roll tests were terminated and leached residues were filtered, washed, dried, weighed, and assayed for gold and silver. The reagent consumption and gold results are presented in the following Table 13-3.

Table 13-3: Drill Core Composites Bottle Roll Tests

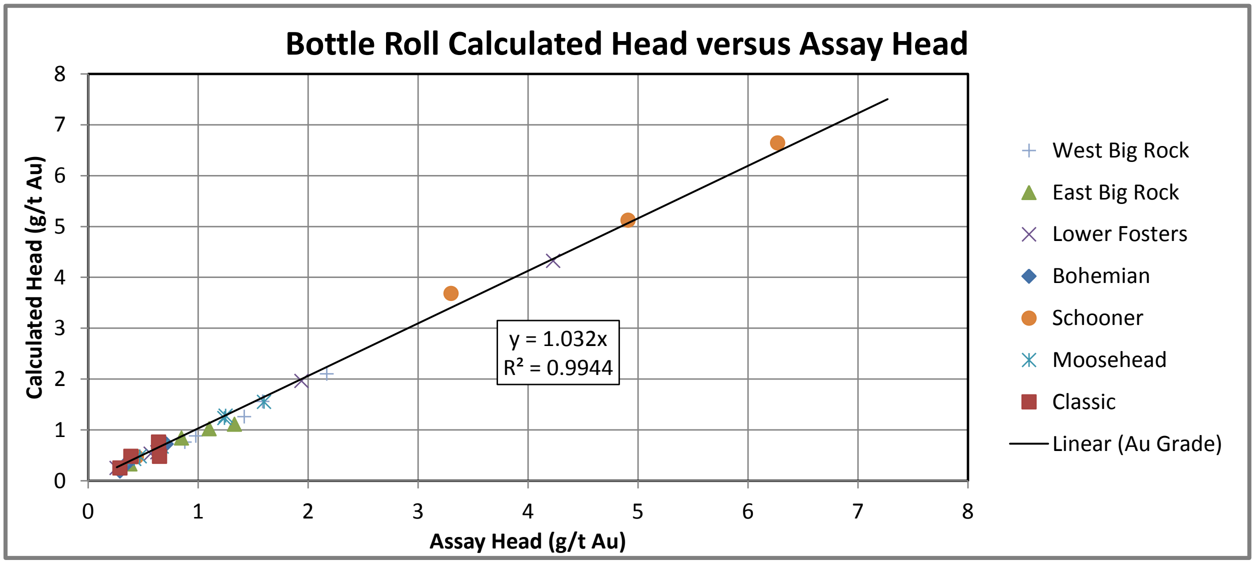
Metallurgical Tests		Gold Grade		Reagent Consumption		Gold Extraction (%)
Ore Zone	Composite I.D.	Calculated Head Assay (g/t)	Tail Assay (g/t)	NaCN (kg/t)	Lime (kg/t)	
	BC12-01	2.10	0.14	0.31	5.30	93

Table 13-3: Drill Core Composites Bottle Roll Tests

Metallurgical Tests		Gold Grade		Reagent Consumption		Gold Extraction (%)
Ore Zone	Composite I.D.	Calculated Head Assay (g/t)	Tail Assay (g/t)	NaCN (kg/t)	Lime (kg/t)	
West Big Rock	BC12-02	0.76	0.31	0.34	3.10	59
	BC12-03	1.26	0.18	0.35	4.30	86
	BC12-04	0.88	0.82	0.45	4.20	7
	BC12-27	1.56	0.07	0.25	4.40	96
East Big Rock	BC12-05	1.11	0.51	0.36	4.00	54
	BC12-06	0.84	0.11	1.56	4.00	87
	BC12-07	0.33	0.07	0.16	3.30	79
	BC12-08	1.02	0.45	0.46	3.70	56
Lower Fosters	BC12-09	0.25	0.10	0.37	3.60	60
	BC12-10	0.53	0.14	0.23	2.60	74
	BC12-11	0.56	0.09	0.08	3.20	84
	BC12-12	1.96	0.34	0.20	4.40	83
	BC12-13	4.32	3.84	1.27	4.50	11
Bohemian	BC12-14	0.19	0.04	0.30	3.80	79
	BC12-15	0.33	0.04	0.35	2.90	88
	BC12-16	0.72	0.09	0.28	3.10	88
	BC12-17	0.35	0.02	0.29	3.50	94
Schooner	BC12-18	0.47	0.02	<0.07	2.50	96
	BC12-19	5.12	0.77	0.28	3.10	85
	BC12-20	3.68	0.72	0.14	2.80	51
	BC12-28	6.64	3.55	0.35	2.80	47
Moosehead	BC12-21	1.28	1.22	0.49	1.60	5
	BC12-22	0.67	0.32	0.30	2.40	52
	BC12-23	0.43	0.37	0.78	1.90	14
	BC12-24	1.23	1.03	0.40	2.40	16
	BC12-25	0.48	0.48	0.72	1.80	0
	BC12-26	1.55	1.55	0.65	1.80	0
Classic	BC12-29	0.25	0.01	0.18	2.50	96
	BC12-30	0.48	0.24	0.55	3.70	50
	BC12-31	0.76	0.19	0.19	2.00	75
	BC12-32	0.48	0.26	0.40	4.20	46

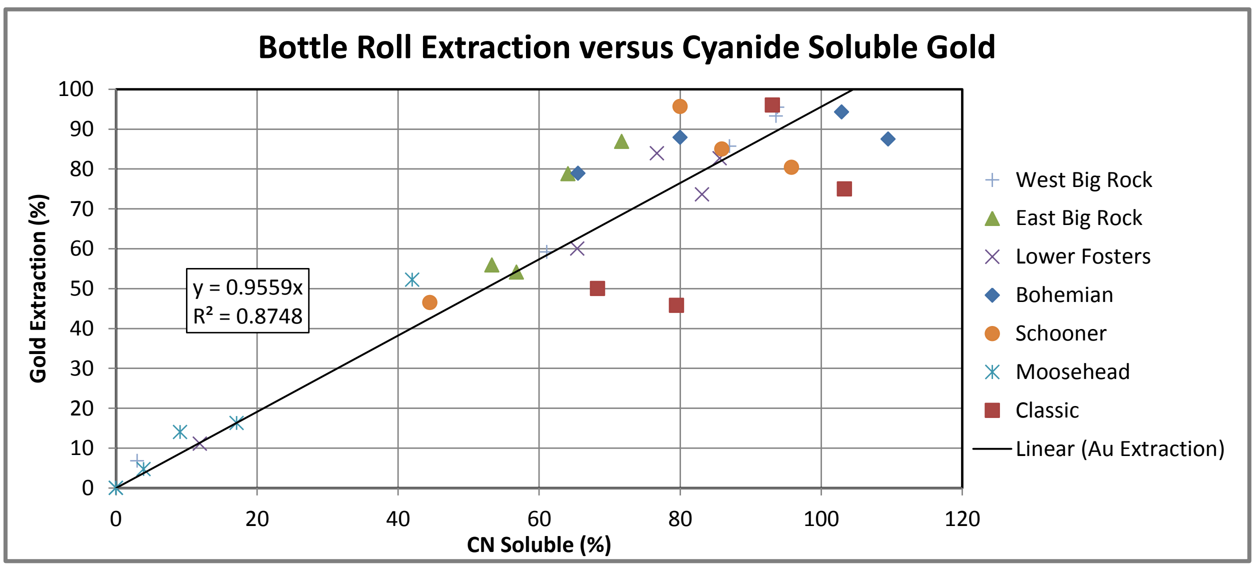
The calculated gold head assays obtained from the bottle roll mass balance check very well with the head assays presented in Table 13-3 as shown in Figure 13–1.

Figure 13–1: Bottle Roll Head Assays



Bottle roll gold extraction varied widely from zero to 96 percent. The ratios of cyanide soluble gold to total, presented in Table 13-3 reasonably predict the variation in bottle roll gold extraction as shown in the following Figure 13–2.

Figure 13–2: Bottle Roll Gold Extraction Results



13.2.3 Locked Cycle Column Leach Testing

A locked cycle column leach test program was conducted on 80 percent passing 9.5 mm composite samples, a total of 32 column tests were completed at the end of the test program. Each test charge weighed approximately 71 kg, with the exception of one test using a 34 kg test charge, and was either not agglomerated or agglomerated with either lime or cement and water to optimum moisture content before loading in the column.

Feed solution containing 1.0 g/l NaCN was applied at a rate of 0.005 gpm/ft^2 and pregnant leach solutions volumes were measured daily and samples were collected and submitted for gold and silver analysis, pH and cyanide concentration. The pregnant leach solutions were pumped through a three stage carbon circuit for adsorption of dissolved gold values. Barren solution, with appropriate make-up reagents, was applied to the ore charges daily.

After the leach-rest cycles, wash cycles were conducted to remove residual cyanide and to recover dissolved gold values. At the end of the wash cycles, drain cycles were conducted to remove excess solution from the residues. After leaching, washing, and draining, residues are unloaded from the columns, air dried, blended and split to obtain samples for a tail assay analyses.

Column test physical are shown in Table 13-4. This table can also be seen in the McClelland Report, Table 109. The difference between the two tables is the saturated moisture. In the McClelland report it is described as dry basis. In this report the saturated moisture is calculated on a wet basis so that retained moistures and agglomeration moistures can be directly compared.

Table 13-4: Drill Core Composites Column Leach Tests Physical Characteristics

Ore Zone	Comp. ID	Test No.	Charge (kg)	Passing 100 mesh (%)	Ore Moisture (wt. %), wet basis				Apparent Bulk Density (t/m³)	
					As Rec'd	For Aggl.	To Saturate	Retained	Before	After
West Big Rock	BC12-01	P-1	71.48	6.0	0.5	0.5	9.1	9.4	1.35	1.39
	BC12-02	P-2	70.96	7.0	0.4	0.4	8.7	8.5	1.37	1.40
	BC12-03	P-3	71.75	4.3	0.4	0.4	8.9	6.1	1.46	1.50
	BC12-04	P-4	71.79	6.0	0.0	0.0	11.7	7.0	1.36	1.38
	BC12-27	P-27	71.24	4.5	0.6	7.6	14.6	7.6	1.30	1.28
East Big Rock	BC12-05	P-5	71.54	8.4	0.6	0.6	10.1	10.8	1.40	1.42
	BC12-06	P-6	33.79	7.4	0.4	0.4	7.1	11.3	1.45	1.45
	BC12-07	P-7	71.33	8.3	0.4	0.4	6.4	19.5	1.47	1.48
	BC12-08	P-8	71.27	3.7	0.4	0.4	8.5	8.0	1.46	1.53
Lower Fosters	BC12-09	P-9	71.23	18.8	0.3	9.7	19.8	9.5	1.25	1.43
	BC12-10	P-10	71.49	6.1	0.2	0.2	7.3	8.6	1.36	1.48
	BC12-11	P-11	68.81	7.0	0.3	0.3	7.7	10.2	1.29	1.38
	BC12-12	P-12	71.51	10.3	0.3	10.0	17.0	10.9	1.23	1.28
	BC12-13	P-13	71.52	4.1	0.1	0.1	6.7	6.3	1.42	1.43
Bohemian	BC12-14	P-14	71.83	4.3	0.0	6.7	13.3	6.3	1.38	1.38
	BC12-15	P-15	71.90	3.4	0.0	5.3	12.5	5.8	1.41	1.42
	BC12-16	P-16	71.89	6.3	0.0	6.6	12.9	6.8	1.35	1.37
	BC12-17	P-17	71.97	4.8	0.0	6.6	14.2	8.7	1.36	1.38
Schooner	BC12-18	P-18	71.91	5.1	0.0	6.2	14.0	7.0	1.36	1.37
	BC12-19	P-19	71.72	3.5	0.0	5.2	12.3	6.7	1.35	1.37
	BC12-20	P-20	71.82	5.0	0.0	6.4	12.7	8.1	1.34	1.37
	BC12-28	P-28	71.60	4.1	0.3	5.4	18.3	6.7	1.41	1.43
Moosehead	BC12-21	P-21	71.53	4.9	0.2	0.2	6.3	6.0	1.47	1.51
	BC12-22	P-22	71.74	4.8	0.4	0.4	6.3	7.4	1.39	1.44
	BC12-23	P-23	71.40	2.8	0.3	0.3	6.2	5.9	1.37	1.42
	BC12-24	P-24	71.42	2.7	0.2	0.2	6.6	6.9	1.38	1.42
	BC12-25	P-25	71.42	4.0	0.3	0.3	7.2	6.5	1.39	1.45
	BC12-26	P-26	71.32	4.3	0.3	0.3	6.3	3.4	1.47	1.50

Table 13-4: Drill Core Composites Column Leach Tests Physical Characteristics

Ore Zone	Comp. ID	Test No.	Charge (kg)	Passing 100 mesh (%)	Ore Moisture (wt. %), wet basis				Apparent Bulk Density (t/m³)	
					As Rec'd	For Aggl.	To Saturate	Retained	Before	After
Classic	BC12-29	P-29	71.08	3.2	1.0	7.3	13.1	10.6	1.44	1.46
	BC12-30	P-30	71.39	3.8	0.6	6.1	19.0	5.5	1.43	1.44
	BC12-31	P-31	71.30	5.4	0.8	6.9	12.4	6.5	1.59	1.63
	BC12-32	P-32	71.01	5.8	1.0	6.8	11.9	6.8	1.44	1.44

Based on the results of the head screen analyses, agglomerate strength and stability tests and bottle roll leach tests, the column leach test charges were either blended with lime and loaded into columns at the as received moisture content, agglomerated with lime and loaded into columns, or agglomerated with cement and loaded into columns. Agglomeration with cement was conducted on two columns with size distributions containing greater than 10 percent minus 100 mesh material, both from Lower Fosters. The column test data logs did not note any problems with solution permeability at the targeted irrigation rate of 0.005 gpm/ft^2. All column test charges were loaded into 6-inch diameter columns, with the exception of P-6, which used a 4-inch diameter column, to an initial height of approximately 10 feet. As received moisture contents were consistently low at 1 percent, or less. Moisture under leach (to saturate) varied widely from 6 to 20 percent. Retained moisture values also varied widely with one unexpected result in Test P-7 where the retained moisture is three times greater than the saturation moisture. Initial dry bulk densities ranged from 1.3 to 1.6 t/m^3. Similarities in characteristics are apparent for each pit except Lower Fosters, where more variation in moisture and particle distribution is seen between composites.

Table 13-5: Drill Core Composites Column Leach Tests

Metallurgical Tests			Gold Head Grade (g/t)		Reagent Consumption			Gold Extraction	
Ore Zone	Composite I.D.	Test No.	Assay	Calculated	NaCN (kg/t)	Lime (kg/t)	Cement (kg/t)	Indicated (%)(1)	Final (%)
West Big Rock	BC12-01	P-1	2.15	2.13	0.99	4.8	-----	94.4	95.3
	BC12-02	P-2	0.84	0.82	0.99	2.8	-----	73.8	75.6
	BC12-03	P-3	1.36	1.43	0.99	3.9	-----	91.8	87.4
	BC12-04	P-4	0.96	0.92	1.24	3.8	-----	24.0	25.0
	BC12-27	P-27	1.58	1.58	0.75	4.0	-----	96.8	96.8
East Big Rock	BC12-05	P-5	1.28	1.28	1.68	3.6	-----	76.6	76.6
	BC12-06	P-6	0.85	0.83	1.23	3.6	-----	87.1	89.2
	BC12-07	P-7	0.36	0.37	1.06	3.0	-----	88.9	86.5
	BC12-08	P-8	1.08	1.07	0.86	3.3	-----	72.2	72.9
Lower Fosters	BC12-09	P-9	0.25	0.23	0.89	-----	6.0	67.0	73.9
	BC12-10	P-10	0.59	0.56	0.77	2.3	-----	80.7	82.1
	BC12-11	P-11	0.61	0.55	0.68	2.9	-----	73.6	81.8
	BC12-12	P-12	1.91	1.90	0.87	-----	8.0	92.7	93.2
	BC12-13	P-13	4.32	4.45	1.82	4.1	-----	12.7	12.4
Bohemian	BC12-14	P-14	0.26	0.22	0.81	3.4	-----	61.5	72.7
	BC12-15	P-15	0.34	0.33	0.74	2.6	-----	84.8	87.9
	BC12-16	P-16	0.71	0.79	0.75	2.8	-----	91.2	82.3

Table 13-5: Drill Core Composites Column Leach Tests

Metallurgical Tests			Gold Head Grade (g/t)		Reagent Consumption			Gold Extraction	
Ore Zone	Composite I.D.	Test No.	Assay	Calculated	NaCN (kg/t)	Lime (kg/t)	Cement (kg/t)	Indicated (%)(1)	Final (%)
	BC12-17	P-17	0.35	0.35	0.74	3.2	-----	91.0	91.4
Schooner	BC12-18	P-18	0.44	0.46	0.64	2.3	-----	97.1	93.5
	BC12-19	P-19	5.00	5.07	1.54	2.8	-----	91.4	90.1
	BC12-20	P-20	3.44	3.50	1.44	2.5	-----	84.9	83.4
	BC12-28	P-28	6.47	6.98	0.66	2.5	-----	48.1	44.6
Moosehead	BC12-21	P-21	1.27	1.29	0.61	1.5	-----	1.6	1.6
	BC12-22	P-22	0.67	0.61	0.76	2.2	-----	44.8	49.2
	BC12-23	P-23	0.42	0.39	0.67	1.7	-----	9.5	10.3
	BC12-24	P-24	1.24	1.17	0.54	2.1	-----	11.3	12.0
	BC12-25	P-25	0.48	0.47	0.91	1.6	-----	0.0	0.0
	BC12-26	P-26	1.59	1.65	0.46	1.6	-----	0.0	0.0
Classic	BC12-29	P-29	0.27	0.22	0.39	2.3	-----	77.8	95.5
	BC12-30	P-30	0.60	0.58	1.44	3.3	-----	40.0	41.4
	BC12-31	P-31	0.69	0.68	0.96	1.8	-----	81.2	82.4
	BC12-32	P-32	0.43	0.41	1.24	3.8	-----	41.9	43.9

Notes: (1) Calculated based on average head assays.

Calculated and assay heads show good agreement. Indicated extraction is calculated based on the cumulative solution analysis and measured head. The final extraction is calculated based on the cumulative solution assays and leached residue assays. Both measurements show good agreement with the average extractions shown in the coarse bottle tests and cyanide shake tests.

The sodium cyanide consumption ranged approximately from 0.61 to 1.82 kg/t. Consumption appears to be material dependent with the pits showing similar consumptions.

Lime was not added for pH control to any of the columns while under leach and pregnant leach solution pH values were acceptable between 10 and 11 for a majority of the time with occasional early periods of 11 to 12, and isolated occurrences of pH greater than 12 or less than 9 at solution breakthrough.

Additional details for the variability test program performed at McClelland can reviewed in the report dated 3 July entitled “Brewery Creek Variability Metallurgical Testing”.

The column leach study contained samples that were not considered potential ore but were tested to observe the metallurgical response. This included samples outside of the proposed pit limits as well as samples from areas with known preg-robbing or refractory response as described by Brewery Creek geology personnel. The results of these column tests were omitted in estimation of gold recovery and reagent consumption by pit. The omitted column tests are:

- West Big Rock BC12-03 Composite Sample
- West Big Rock BC12-04 Composite Sample
- East Big Rock BC12-05 Composite Sample

- Lower Foster BC12-12 Composite Sample
- Lower Foster BC12-13 Composite Sample
- Moosehead BC12-21 Composite Sample
- Moosehead BC12-24 Composite Sample
- Moosehead BC12-25 Composite Sample
- Moosehead BC12-26 Composite Sample

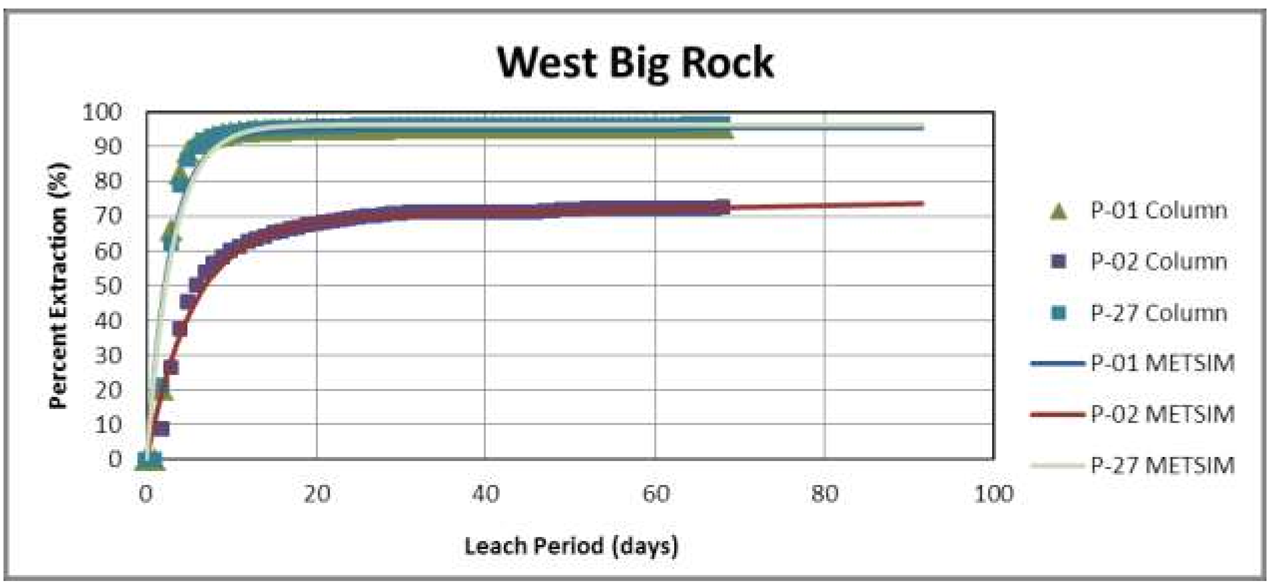
Due to poor metallurgical response from the Moosehead samples, this deposit has been removed from mine planning and had not been considered as part of this PEA. The Classic deposit remains under development and it also is not included in the mine plan or this PEA. Three deposits are included in the mine plan which have not yet been subjected to metallurgical testing, Golden, Kokanee and Lucky. Metallurgical response for these three deposits has been estimated as described in Section 13.4.

The following graphs shown in Figure 13–3 through Figure 13–7 depict the actual column leach data and METSIM projections for gold leach extraction versus leach day. Only the column leach tests used for estimating project gold extraction levels are shown. The METSIM projections fit the extraction data to the following equation in which "t" is leach day and "A1", "A2", "R1" and "R2" are constants derived from the METSIM software.

$$Extraction = (A1 * (1-(1-R1)^{t}) + (A2 * (1-(1-R2)^{t})$$

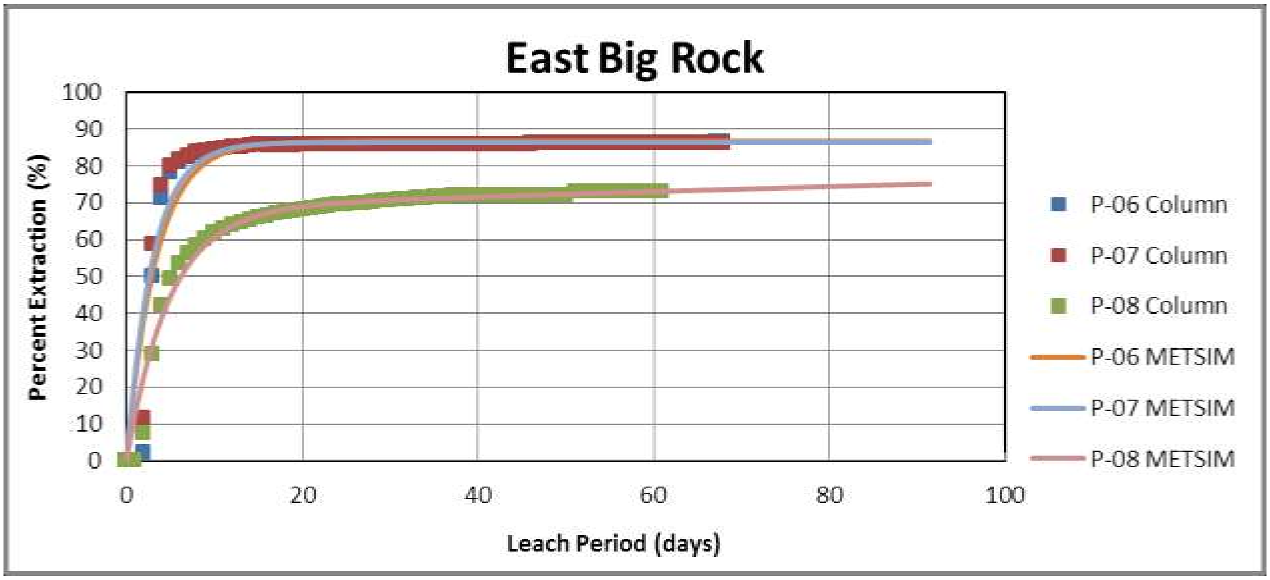
As shown, these equations model the column test data well and facilitate averaging of column test data.

Figure 13–3: West Big Rock Column Leach Gold Extraction



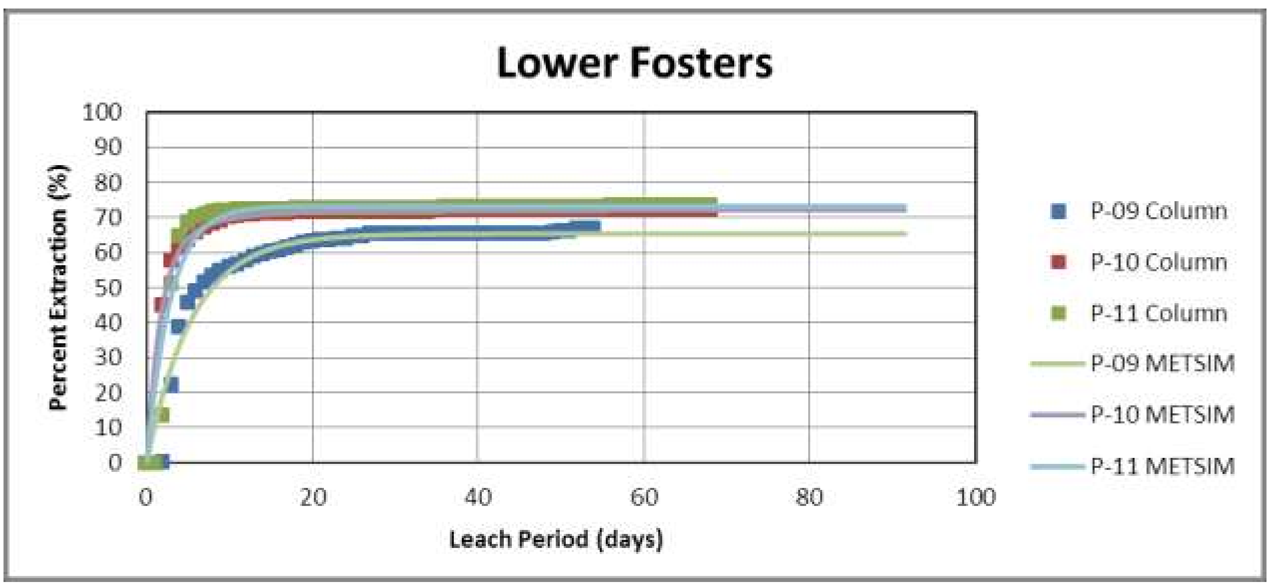
Two columns from West Big Rock attained 95 percent gold extraction and the third 74 percent.

Figure 13–4: East Big Rock Column Leach Gold Extraction



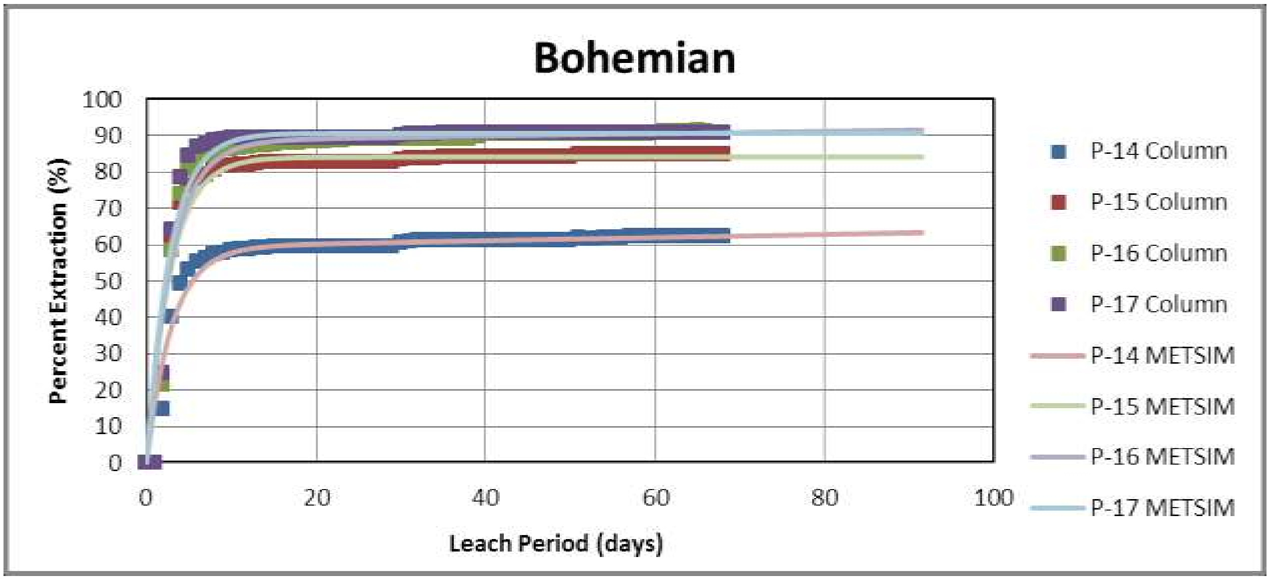
Two columns from East Big Rock attained 87 percent gold extraction and the third 72 percent.

Figure 13–5: Lower Fosters Column Leach Gold Extraction



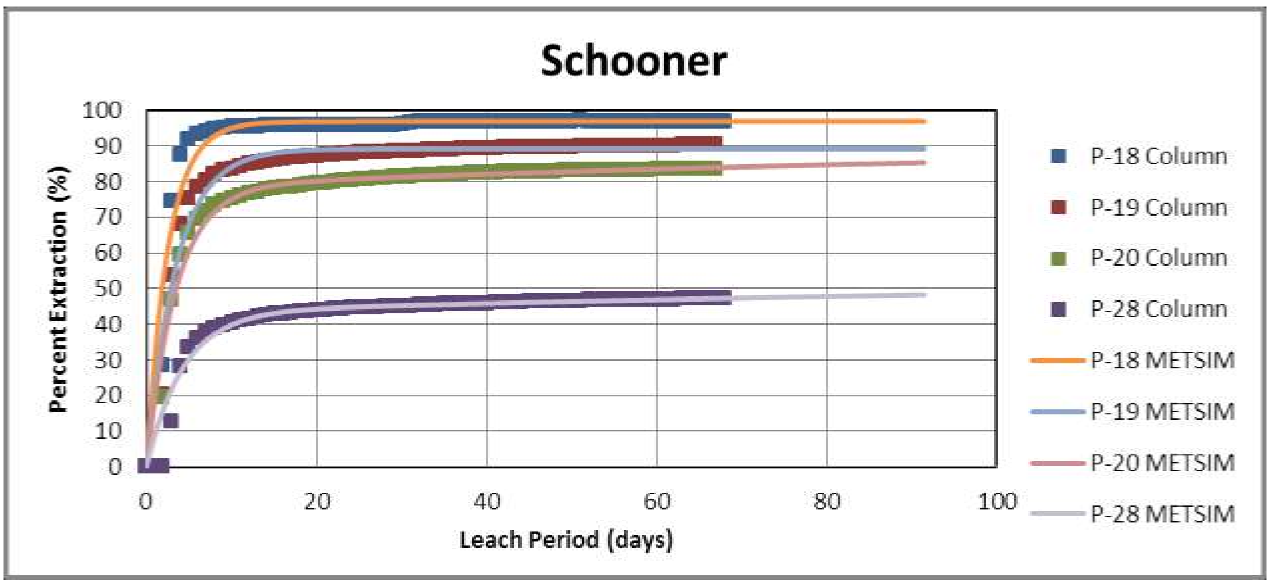
The three Lower Fosters columns attained gold extraction levels around 70 percent though P-11 exhibited a slower rate of leaching.

Figure 13–6: Bohemian Column Leach Gold Extraction



Gold extraction from the four Bohemian columns ranged from 62 to 91 percent.

Figure 13–7: Schooner Column Leach Gold Extraction



Gold extraction from the four Schooner columns ranged from 47 to 97 percent.

In all of the column tests, gold extraction was essentially complete after 30 days of leaching. Under the column test leach conditions, 30 days of leaching corresponds to an approximate cumulative leach solution to ore ratio of 2.0 kl/t.

13.2.4 Prediction of Gold Production

The column test work was conducted in columns with the height of approximately 10 feet of ore depth, which is shorter than the 8 metre lift heights proposed for the heap leach operation. The column leach tests were conducted at an approximate application rate of 5 gpm/ft^2 (12.2 l/h/m^2) which is slightly higher than the proposed 12 l/h/m^2 for the operation. The initial dry bulk density of the columns ranged from 1.3 to 1.6 t/m^3. A constant value of 1.6 t/m^3 has been proposed for the industrial operation. An intermediate leach solution pond has been proposed for the industrial operation, which will allow stacking of leach solutions to increase pregnant leach solution grade. The proposed industrial leach cycle is as follows:

- Fresh ore will be irrigated for 30 days using intermediate leach solution.
- Barren solution will be applied to the ore for an additional 30 days.
- The ore will be buried and leaching will continue for an additional 30 days from leach solution applied to the upper lift.

Under the proposed industrial leaching parameters and leach cycle, the overall industrial cumulative applied leach solution to ore ratio will attain 2.0 kl/t. For scaling purposes, no additional gold extraction is expected to occur when the leach solution to ore ratio exceeds 2.0 kl/t, either as a buried lift or during the 135 days of winter when irrigation continues without the addition of fresh ore to the leach pad.

Table 13-6: Heap Leach Design Parameters

Parameter	Units	Design
Lift Height	m	8
Dry Stack Bulk Density	t/m³	1.6
Solution Application Rate	l/m²/h	12
Leach Stages:		
Primary Leach (ILS Solution Application)	days (S:O ratio)	30 (0.7)
Secondary Leach (Barren Solution Application)	days (S:O ratio)	30 (0.7)
Buried Lift Leach (PLS Application)	days (S:O ratio)	30 (0.7)
Total Leach Cycle	days (S:O ratio)	90 (2.0)

The gold extraction data has been extracted from the column tests when solution to ore ratios attained approximately 0.7, 1.3 and 2.0 kl/t. Gold extraction has been calculated from the METSIM projections at this exact solution to ore ratios. Both of these are shown in the following Table 13-7. Also shown are the average results for each ore zone.

Table 13-7: Column Leach Tests and Modeled Gold Extraction

Metallurgical Tests			Leach Test Gold Extraction (%)			METSIM Projection Gold Extraction (%)		
			Approximate S:O Ratio			S:O Ratio		
Ore Zone	**Composite I.D.**	**Test No.**	**0.7**	**1.3**	**2.0**	**0.7**	**1.3**	**2.0**
West Big Rock	BC12-01	P-1	93.9	95.0	95.1	92.6	95.4	95.4
	BC12-02	P-2	59.8	67.7	70.3	58.5	68.2	70.1
	BC12-27	P-27	93.8	95.5	96.0	92.7	96.2	96.3
	Average		**82.5**	**86.1**	**87.1**	**81.3**	**86.6**	**87.3**
East Big Rock	BC12-06	P-6	84.4	86.0	86.1	82.3	86.5	86.7
	BC12-07	P-7	84.6	85.9	85.9	83.5	86.3	86.4
	BC12-08	P-8	61.8	68.3	70.3	60.6	69.0	70.5

	Average		**76.9**	**80.1**	**80.8**	**75.5**	**80.6**	**81.2**
Lower Fosters	BC12-09	P-9	60.7	67.5	70.9	59.9	68.1	70.4
	BC12-10	P-10	80.0	81.0	81.9	79.9	81.7	81.8
	BC12-11	P-11	79.9	80.4	81.3	78.6	80.4	80.4
	Average		**73.5**	**76.3**	**78.0**	**72.8**	**76.7**	**77.5**
Bohemian	BC12-14	P-14	58.4	59.4	59.4	57.7	60.2	60.6
	BC12-15	P-15	81.7	82.7	82.7	81.8	84.1	84.1
	BC12-16	P-16	86.3	88.3	88.9	85.9	89.1	89.5
	BC12-17	P-17	89.3	89.5	89.5	88.4	90.6	90.7
	Average		**78.9**	**80.0**	**80.1**	**78.5**	**81.0**	**81.2**
Schooner	BC12-18	P-18	95.3	95.8	95.8	95.1	96.9	96.9
	BC12-19	P-19	83.7	87.3	88.4	84.2	88.9	89.2
	BC12-20	P-20	75.3	79.5	81.1	75.1	80.2	81.2
	BC12-28	P-28	40.5	43.9	45.0	39.9	44.3	45.2
	Average		**73.7**	**76.6**	**77.6**	**73.6**	**77.6**	**78.1**

Gold recovery to doré has been estimated for the individual deposits and shown in the following Table 13-8. The average values of the gold extractions from the METSIM projections were used as bases. These levels of extraction have been downgraded by 3.5 percent to reflect the attainable heap leach extraction due to scale up (increased lift height, losses on the side of the heap and channeling effects) on finely crushed ore. Additionally, gold extraction is downgraded by one percent to account for losses in the metal recovery processes.

Table 13-8: Weighted Average for Lime and Cement Additions from Column Tests

	Gold Extraction by Ore Zone (%)				
	West Big Rock	**East Big Rock**	**Lower Foster**	**Bohemian**	**Schooner**
Cumulative Extraction					
30 days (0.7 kl/t)	81.3	75.4	72.8	78.4	73.6
60 days (1.3 kl/t)	86.6	80.6	76.7	81.0	77.6
90 days (2.0 kl/t)	87.3	81.2	77.6	81.2	78.1
Discount for Industrial Practice	3.5	3.5	3.5	3.5	3.5
Heap Leach Average Extraction	83.8	77.7	73.2	77.7	74.6
CIC/Goldroom Recovery	99.0	99.0	99.0	99.0	99.0
Gold Recovery to Doré	**82.9**	**77.0**	**72.5**	**77.0**	**73.9**

13.2.5 Prediction of Reagent Consumption

Lime and cement consumption are estimated based on results from the column leach tests. Consumptions for each deposit are estimated as averages of the column test results. As the column test consumptions were calculated as reagent added prior to loading the columns and no additional lime or cement was added during the leach cycles, these levels of consumption are independent of leach time.

Table 13-9: Weighted Average for Lime and Cement Additions from Column Tests

Ore Zone	**Reagent Consumption (kg/t)**	
	Lime	**Cement**

West Big Rock	3.87	-
East Big Rock	3.30	-
Lower Fosters	1.73	2.00
Bohemian	3.00	-
Schooner	2.53	-

Sodium cyanide consumption from the bottle roll and column leach tests is shown in the following Table 13-10. In all cases the bottle roll consumption was lower than the column leach test consumption. This is in part due to the extended leach cycle of the column tests. Column test sodium cyanide consumption and the pregnant leach solution free sodium cyanide concentration, at the point when the applied leach solution to ore ratio was 2 kl/t, are also shown in the Table 13-10. At a 2 kl/t solution to ore ratio, the overall average cyanide consumption is 0.66 kg/t and range from 0.49 to 0.85 kg/t. All column tests show high free cyanide in the pregnant leach solutions, 0.72 g/l on average, which is higher than would be targeted for an operating heap leach.

Table 13-10: Sodium Cyanide Consumption

Ore Zone	Composite I.D.	Bottle Roll NaCN Consumption (kg/t)	Reported Column Test NaCN Consumption (kg/t)	Column Test at S:O Ratio = 2 kl/t	
				NaCN Consumption (kg/t)	PLS NaCN Concentration (g/l)
West Big Rock	BC12-01	0.31	0.99	0.71	0.85
	BC12-02	0.34	0.99	0.67	0.65
	BC12-27	0.25	0.78	0.49	0.70
	Average	**0.30**	**0.92**	**0.62**	**0.73**
East Big Rock	BC12-06	1.56	1.23	0.85	0.65
	BC12-07	0.16	1.06	0.70	0.65
	BC12-08	0.46	0.86	0.75	0.75
	Average	**0.73**	**1.05**	**0.77**	**0.68**
Lower Fosters	BC12-09	0.37	0.96	0.72	0.75
	BC12-10	0.23	0.77	0.72	0.75
	BC12-11	0.08	0.68	0.75	0.62
	Average	**0.23**	**0.80**	**0.73**	**0.71**
Bohemian	BC12-14	0.30	0.86	0.69	0.75
	BC12-15	0.35	0.82	0.67	0.80
	BC12-16	0.28	0.85	0.57	0.75
	BC12-17	0.29	0.79	0.66	0.70
	Average	**0.31**	**0.83**	**0.65**	**0.75**
Schooner	BC12-18	<0.07	0.72	0.58	0.85
	BC12-19	0.28	1.37	0.62	0.60
	BC12-20	0.14	1.26	0.49	0.85
	BC12-28	0.35	1.37	0.56	0.65
	Average	**0.26**	**1.18**	**0.56**	**0.74**

A review of the historic operational data reveals that in the final year of operation, 2000, Brewery Creek consumed 0.34 kg/t sodium cyanide with a plant to date consumption of 0.21 kg/t (Viceroy, 2000). As these levels of consumption are more in line with the levels of consumption observed in the bottle roll tests, the average consumptions obtained from the bottle roll tests are used to estimate sodium cyanide consumption by deposit.

Tests were not conducted to determine if the gold extraction was sensitive to sodium cyanide concentration.

13.3 MLI Job No. 3618 Reprocessing of Original Heap

The original project at Brewery Creek processed an estimated 9.5 million tonnes of ore on a truck dumped heap leach. This ore is scheduled to be reprocessed along with the new ore. A drilling program was conducted on the old heap and 18 sonic drillholes were drilled down to a maximum depth of 25 metres. One hundred-seventy seven head samples were analyzed and composited into 28 samples for bottle roll analysis. The individual holes were then combined into 4 composite samples, RZ-1 to 4, and crushed to 9.5 mm, agglomerated, and column leached (SGS, 2012).

Each drill core interval sample was analyzed for total gold and silver, cyanide soluble gold and silver, and preg-robbing potential of the carbonaceous minerals. Composite samples were reconstituted from the interval samples and subjected to bottle roll leach testing, vat leach testing and column leach testing. Bottle roll testing and column leach testing were conducted on composites prepared by ore zones to evaluate re-handling and crushing the residue material. Screen analyses were conducted on column composite sample heads and residual tails. Vat leach tests included fresh water rinsing followed by cyanide leaching to evaluate rinsing or re-leaching the existing residue material.

13.3.1 Reprocessed Ore Sample Preparation

A total of 177 drill core interval samples were delivered to McClelland on 21 October 2011 from the Brewery Creek Property. A majority of the drill core intervals were 1.5 metres long and weighed approximately 20 kg. There were 16 interval samples measured approximately 1 metre in length and weighed between 2 kg and 20 kg. The interval samples were prepared according to the following procedures to generate test charges for the interval head assays and composite head assays;

- Each drill core interval sample was dried and the dry samples with 8 kg or more blended and split in half by coring and quartering methods.
- The interval samples with less than 8 kg of weight stage crushed to 80 percent passing 38 mm (100 percent passing 50 mm) prior to splitting.
- One half of the interval sample prepared in the previous stages was saved for further tests. The other half was stage crushed to 80 percent passing 9.5 mm (100 percent passing 12.5 mm).
- A 1,000 gram was split from each 80 percent passing 9.5 mm sample and submitted for the interval sample Au and Ag assays.
- The un-crushed and 80 percent passing 38 mm materials prepared in the first two stages were combined to generate a total of 28 drillhole composite samples representing the upper and lower portions of the drillholes.
- The 60 to 60.6 feet interval from the drillhole BCS 6-2 was not included in these composites. It was suspected by the client that this 0.6 foot long interval had been mislabelled and/or included by mistake.
- A 15 kg split was taken from each composite for vat leach tests.
- The remaining composite samples were crushed to 80 percent passing 38 mm (100 percent passing 50 mm), blended and a 5 kg split was obtained by coning and quartering methods.

- Each 5 kg composite was stage crushed to 80 percent passing 9.5 mm and 1,000 gram test charges were split for composite head assays. The composite head assay analyses were conducted in triplicate.
- The following paragraphs describe the steps used to generate composite test charges for stability tests, bottle roll testing and column leach testing;
- Composite samples prepared in the previous steps were blended by ore zone and test charges were split for agglomerate strength and stability tests (10 kg) and bottle roll testing (5 kg).
- The 5 kg split was crushed to 80 percent passing 9.5 mm (100 percent passing 12.5 mm) and 1,000 gram test charges were split for bottle roll testing and head assay analysis. Head assays were conducted in triplicate using 1,000 gram splits.
- The reject 38 mm material from each residue zone composites was blended and 90 kg splits were stage crushed to 80 percent passing 9.5 mm (100 percent passing 12.5 mm). From the 9.5 mm material 75 kg was split for column leach tests and the remaining material (~15 kg) was used for head screen analyses.

13.3.2 Reprocessed Ore Column Leach Tests

Head assay splits from each interval and the composites were assayed for gold and silver content by conventional fire assay and geochemical methods. The cyanide soluble gold and cyanide soluble silver were determined by cyanide shake tests. Details of gold and silver assay results for the interval samples and the composite samples are shown in the appendix of the McClelland report. Physical characteristics for the column leach tests are shown in the following Table 13-11.

Table 13-11: Reprocessed Ore Physical Characteristics

Sample Designation	Test No.	Ore Charge (kg)	Passing 100 mesh (%)	Moisture, (Weight %)				Apparent Bulk Density (t/m³)	
				As Rec'd.	for Aggl.	to Saturate	Retained	Before	After
RZ-1	P-1	71.69	16.9	0.3	8.9	18.0	10.4	1.28	1.35
RZ-2	P-2	69.96	16.8	0.3	9.1	17.1	12.6	1.24	1.29
RZ-3	P-3	72.06	14.6	0.2	8.6	17.5	12.0	1.23	1.27
RZ-4	P-4	70.95	16.2	0.2	8.8	16.0	9.8	1.20	1.24

The high percentage of minus 100 mesh material indicates that cement should be used for agglomeration of the reprocessed material.

The results of the column leach tests conducted on the reprocessed ore are shown in the following Table 13-12.

Table 13-12: Reprocessed Ore Column Leach Tests

Composite I.D.	Gold Extraction (%)	Reagent Consumption	
		NaCN (kg/t)	Cement (kg/t)
RZ-1	42.9	1.54	5.75
RZ-2	54.7	1.80	5.75
RZ-3	53.6	1.87	5.75
RZ-4	57.3	1.89	5.75

The average gold extraction of 49.6 percent has been decreased to a proposed 45.0 percent for heap leaching to account for the idealized conditions in a column leach test and ADR and refinery recovery. Cement consumption for the industrial heap leach is 5.75 kg/t. Sodium cyanide consumption was based on the original heap leach consumption rate noted in the Brewery Creek monthly reports and equal to the average consumption in the last year of operation x 1.25 safety factor, 0.212 x 1.25 = 0.265 kg/t, where new ore was still being placed on the heap (Viceroy, 2000).

Because of the previous issues at Brewery Creek, reprocessing will require sampling before crushing to eliminate any preg-robbing materials. It is estimated that this operational test work will eliminate some of the tonnage on the old pad, and some of the tonnage will be left as a cushion layer to prevent damage to the old pad. Additionally, the volume available on the old and new cells combined limits the amount of tonnage processed from the old pad. An estimated 3.2 million tonnes of material can be reclaimed from the old heap with a total gold grade of 0.71 g/t and processed according to the parameters shown in Table 13-13.

Table 13-13: Reprocessing Parameters from Spent Material

Description	Value	Units	Comments
Recovery	45	%	Test work
Lime Consumption	0	kg/t	Column Leach Tests
Cyanide Consumption	0.265	kg/t	Operational data
Cement Consumption	5.75	kg/t	Column Leach Tests

13.4 Gold Extraction and Reagent Consumption

The following Table 13-14 summarizes the levels of gold extraction and reagent consumption for each of the pit deposits and the reprocessed heap. Included are three deposits, Golden, Kokanee and Lucky, which were not tested by McClelland or SGS, though some historic Viceroy bottle roll test results have been reviewed. For these three deposits, a gold recovery to doré estimate of 70 percent has been assigned. Reagent consumption for these deposits has been estimated as the average of the consumption for West Big Rock, East Big Rock, Lower Fosters, Bohemian and Schooner.

Table 13-14: Weighted Average Extraction and Reagent Consumption

Ore Zone	Tonnes	Au Grade (g/t)	Au Extraction (%)	NaCN (kg/t)	Lime (kg/t)	Cement (kg/t)
West Big Rock	809,000	1.17	82.9	0.30	3.87	0
East Big Rock	465,000	1.07	77.0	0.73	3.30	0
Lower Fosters	1,275,000	1.62	72.5	0.23	1.73	2.00
Bohemian	1,577,000	1.22	77.0	0.31	3.00	0
Schooner	1,044,000	2.07	73.9	0.26	2.53	0
Golden	878,000	1.34	70.0	0.37	2.89	0.40
Kokanee	1,243,000	1.06	70.0	0.37	2.89	0.40
Lucky	2,973,000	1.27	70.0	0.37	2.89	0.40
Old Heap	3,194,000	0.77	45.0	0.27	0	5.75
Total or Weighted Average	13,458,000	1.21	68.9	0.32	2.15	7.71

The weighted average gold extraction and reagent consumption have been used as basis for the process plant design criteria. The reagent consumption values by deposit have been used to estimate overall annual per tonne operating cost. The gold extraction results by deposit have been used in conjunction with the mine plan to estimate gold production.

13.5 Crushing Work Index and Abrasion

Samples of drill core were submitted to Phillips Enterprises LLC for Bond crushing work index and abrasion index testing. The results are summarized below.

Table 13-15: Work Index and Abrasion Results

Ore Zone	Sample ID	Crushing Work Index		Abrasion
		(kW-hr/mt)	kW-hr/short ton	Index
West Big Rock	WBR	4.96	4.50	0.0908
East Big Rock	EBR	5.33	4.83	0.0390
Lower Fosters	LF	9.82	8.91	0.0308
Bohemian	BOH Comp 14-17	12.97	11.76	0.0391
Moosehead	Comp 21-23	15.42	13.99	0.0434
Moosehead	Comp 24-36	13.03	11.82	0.0371

Work and abrasion indices are used to size crushing machines and to estimate the wear incurred during operations.

14.0 MINERAL RESOURCE ESTIMATES

A total of fifteen Mineral Resource estimates are presented in this report which have been prepared or validated independently by three independent Qualified Persons (QP); Don Hulse, P.E., of Gustavson, Michael J. Lechner, P.Geo., of RMI and James Barr, P.Geo., of Tetra Tech EBA.

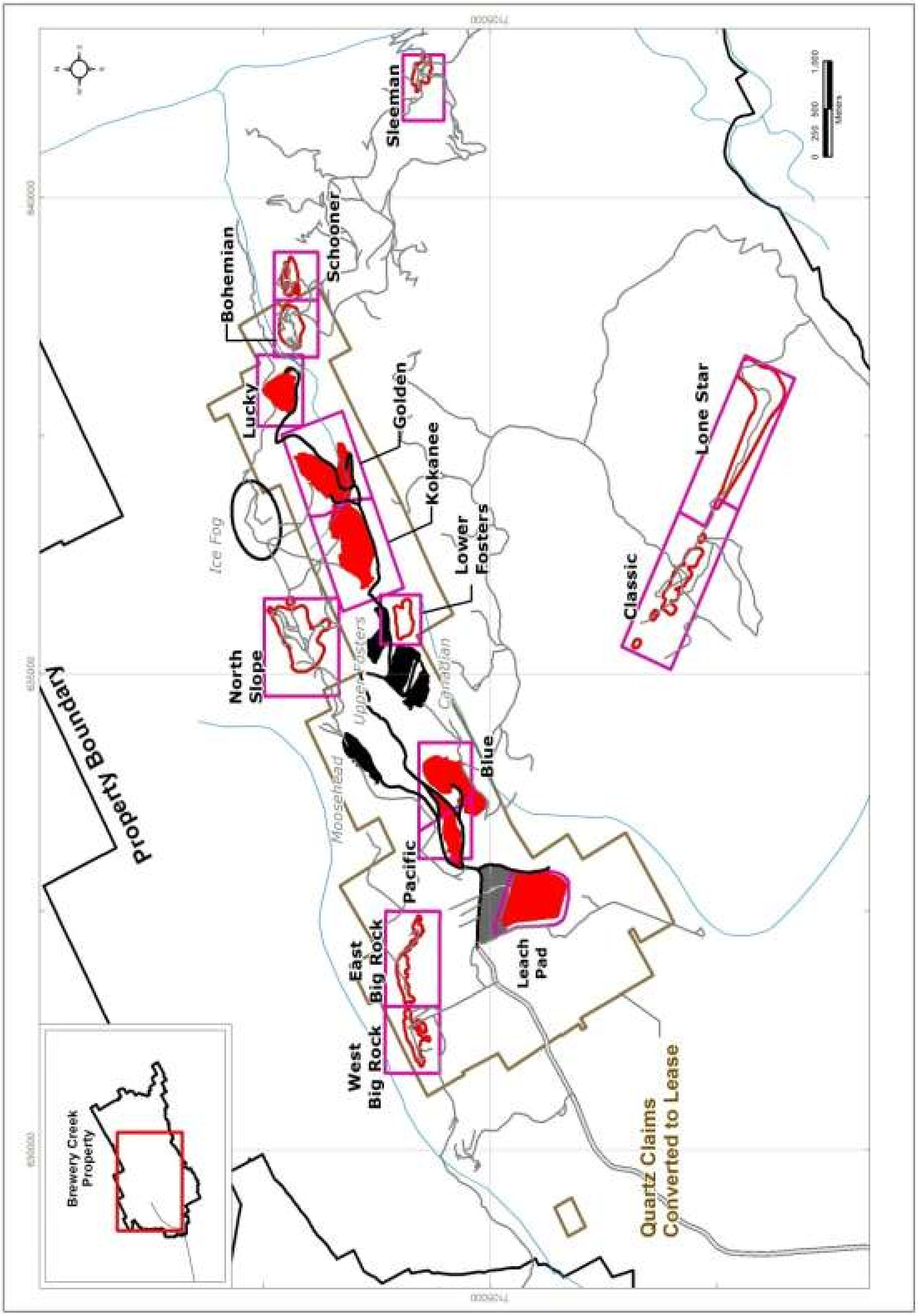
Mr. Hulse is the responsible QP for the Kokanee-Golden (KOGD), Pacific-Blue (PABL), and Lucky (LU) resource estimates. These mineral resource estimates were prepared by Golden Predator and were verified by and approved by Gustavson. These estimates are presented in Sections 14.1, 14.5, and 14.6.

Mr. Lechner prepared the mineral resource estimates for the Bohemian (BH), Schooner (SC), Lower Fosters (FS), West Big Rock (WB), East Big Rock (EB), Classic (CL), and Lone Star (LS) deposits, and for the historical heap leach pad material. Mr. Lechner is the responsible QP for the BH, SC, FS, WB, EB, CL, LS and heap leach pad resource estimates. These estimates are presented in Sections 14.2, 14.3., 14.5, and 14.6.

Mr. Barr prepared the mineral resource estimates for the North Slope (NS) and Sleeman (SL) deposits which were reported as part of the 2013 Technical Report titled “Updated mineral Resources Estimate for the Brewery Creek Property” prepared for Golden Predator Corp., effective March 11, 2012, and amended on January 17, 2013. Mr. Barr is the responsible QP for the NS and SL resource estimates. These estimates are summarized in Sections 14.4, 14.5, and 14.6.

All mineral resource estimates included in this current report were previously reported by Gustavson in the Technical Report titled “NI 43-101 Technical Report on Resources, Brewery Creek Project, Yukon, Canada” effective June 1, 2013 and released October 23, 2013. The resource estimates are unchanged and remain current. The key assumptions, parameters and methods used to calculate the resources have been included herein.

Figure 14–1: Locations of the fifteen block models as purple polygons



14.1 Kokanee, Golden, Pacific, Blue, and Lucky Deposits

The mineral resource estimates for the past producing Pacific-Blue (PABL), Kokanee-Golden (KOGD) and Lucky (LU) resource areas were generated by Bruce Otto and Mark Shutty, staff geologists with Golden Predator, and have been independently verified and approved by Gustavson. The Pacific and Blue deposits were not included as part of the PEA as their mineral resource estimates did not meet the current objectives of Golden Predator.

Drilling in the above mentioned resource areas consists of 1,327 core and RC drillholes which Gustavson is of the opinion provide sufficient data on which to classify a mineral resource estimate as Indicated or Inferred.

14.1.1 Deposit Geology Pertinent to Resource Estimation

The deposits exhibit characteristics of both epithermal type and intrusive-related gold deposits. Gold mineralization consists of fracture-controlled quartz stockwork in both siliciclastic and intrusive rocks along an east-northeast striking, moderately south dipping structural trend (BCRT). Altered intrusive rocks are typically the preferred host for gold mineralization, however gold mineralization at the Pacific deposit exhibits a strong preference for a siltstone host.

Golden Predator constructed a probabilistic lithology model of each target area based on lithology information from drillhole logs. Logged sample intervals were used to estimate the majority lithology, intrusive (1) or sedimentary (2), throughout each deposit and code these values directly to the block model.

Oxidation generally conforms to surface topography but penetrates deeper along structures into altered intrusive rocks and is also noted deeper in pyritized sedimentary rocks at or near intrusive contacts. Because of the multi-dimensional and somewhat localized occurrence of logged oxidation in drill core/cuttings an all-inclusive RedOx surface was neither practical nor possible to construct. As such, a probabilistic indicator oxide model was constructed to completely capture the complex occurrence of oxide material. The procedure codes all eligible blocks as oxide or sulfide via a simplistic and conservative RedOx surface, constructed to envelope all near surface oxide material, then overprints deeper sulfide coding where intrusive lithologies having a greater than 50% probability of being oxidized are encountered.

Viceroy mined but only partially backfilled and reclaimed shallow pits within all three resource model areas. An ultimate pit surface inherited by Golden Predator from Viceroy was used in conjunction with a comprehensive blast drill database to construct a mined surface. LiDAR data points, acquired in 2011 and 2012 by Golden Predator, were used to generate a current topographic surface. All blocks within the model were coded with a percent (below) topo value. And, blocks residing below the topographic surface but above the mined surface were coded as backfill material, making them ineligible for gold grade estimation.

Golden Predator's models consist of generalized structurally bound, sediment/intrusive lithology models coded to account for oxide/sulfide and backfill material types. Gold estimation was conducted using inverse distance (ID) method and validated with nearest neighbor (NN) method for eligible blocks meeting the criteria of residing below the present topographic surface and having an assigned intrusive or sediment rock type.

14.1.2 Data Used for Estimation

Golden Predator created a 3D block model of the mineral resource based on current and historical data. The resource estimate is divided into three areas consisting of 5 targets. The Kokanee-Golden (KOGD) model consists of the Kokanee (KO) and Golden (GD) targets; the Pacific-Blue (PABL) model consists of the Pacific (PA) and Blue (BL) targets, and the Lucky (LU) model consists of the Lucky (LU) target.

The drillhole database contains 1327 drillholes with assay values that fall within the 3 model areas. Drillhole location detail by resource area is shown in Figure 14–8, Figure 14–11 and Figure 14–13.

A statistical analysis of the drillhole samples is presented in Table 14-1.

Table 14-1: Sample Gold Assay Statistics (Gold Grades Reported in g/t)

Resource Area	Number Samples	Min	Max	Mean	Std. Dev.	Median
KOGD	30675	0.0025	27.36	0.400	1.203	0.05
PABL	10695	0.0001	24.20	0.483	1.416	0.06
LU	6419	0.0025	27.50	0.489	1.516	0.05

14.1.3 Bulk Density

Bulk density (specific gravity, SG) was assigned on a block by block basis and determined by the majority lithology and oxidation state of the block. For blocks that were modeled in the oxide zone, a specific gravity of 2.57 g/cm³ was assigned for both sedimentary and intrusive lithologies. Within the sulfide zone, blocks modeled with an intrusive lithology were assigned a SG of 2.64g/cm³ and blocks modeled with a sedimentary lithology were assigned a SG of 2.67 g/cm³. All blocks that were modeled as backfill were given a SG of 1.8 g/cm³.

14.1.4 Methodology

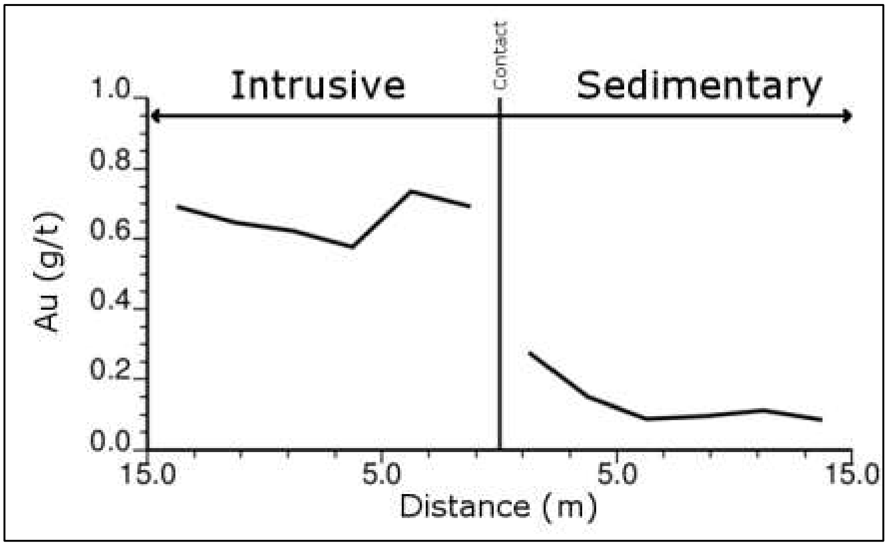
Golden Predator constructed a 3D block model for each of the three resource areas in MineSight ® modeling software. Each resource area was broken down into structural domains to accommodate local anisotropy during estimation. The KOGD area contains 6 structural domains, the PABL area contains 7 structural domains, and the LU area contains 2 structural domains. Search ellipse orientations for these structural domains are specified in the modeling parameters section and Table 14-6.

No discreet higher grade areas were modeled for the resource estimate. The structural and lithologic domains were instead used to constrain estimation. Lithologic domains (LDMN) were used to constrain gold estimation by way of an intrusive or sediment block coding value. In lieu of constructing deterministic 3D wireframes representing the two major rock types, a probabilistic, categorical indicator block modeling method has been used to model lithologic data. The process first uses an ID3 interpolation of uncomposited major rock type values to estimate lithology for all blocks within the defined search ellipse. The lithology type having the highest probability is assigned to each block. A code matching restriction requiring only blocks and samples having like coded lithology was then used in gold estimation, i.e. intrusive samples can only be used to estimate an intrusive coded block.

Contact plots comparing intrusive (1) and sediment (2) major rock categories against the Au variable clearly demonstrate the existence of a distinct boundary between these grouped lithologies and lend support to LDMN stationarity. An example contact plot is shown in Figure 14–2, demonstrating Au affinity for an intrusive host and occurrence of elevated values at the sediment-intrusive contact.

Structural domains (SDMN) were established to distinguish areas having continuity of mineralization, typically within a fault-bound space. Each domain has a unique orientation for the purpose of optimizing search parameters within the model during the estimation procedure. Golden Predator utilized oriented search ellipses based on structural trends within each target and resource area. All estimations were done using an Inverse Distance methodology with a power of 3 (ID3).

Figure 14–2: Lucky Area Contact Plot



All block models used blocks that are 6 metres along strike, 6 metres normal to the structure, and 6 metres high. Each of the blocks was assigned attributes of gold grade, weighted rock density, structural domain (SDMN), and majority rock type. Block model parameters are shown in Table 14-2.

Table 14-2: Block Model Parameters

Resource Area		Origin (UTM m)	Number of Blocks	Block Size (m)	Rotation
KOGD	X	635900	330	6	0
	Y	7105940	120	6	0
	Z	700	70	6	-20
PABL	X	633050	204	6	0
	Y	7105200	97	6	0
	Z	560	60	6	0
LU	X	637600	125	6	0
	Y	7107050	85	6	0
	Z	600	75	6	0

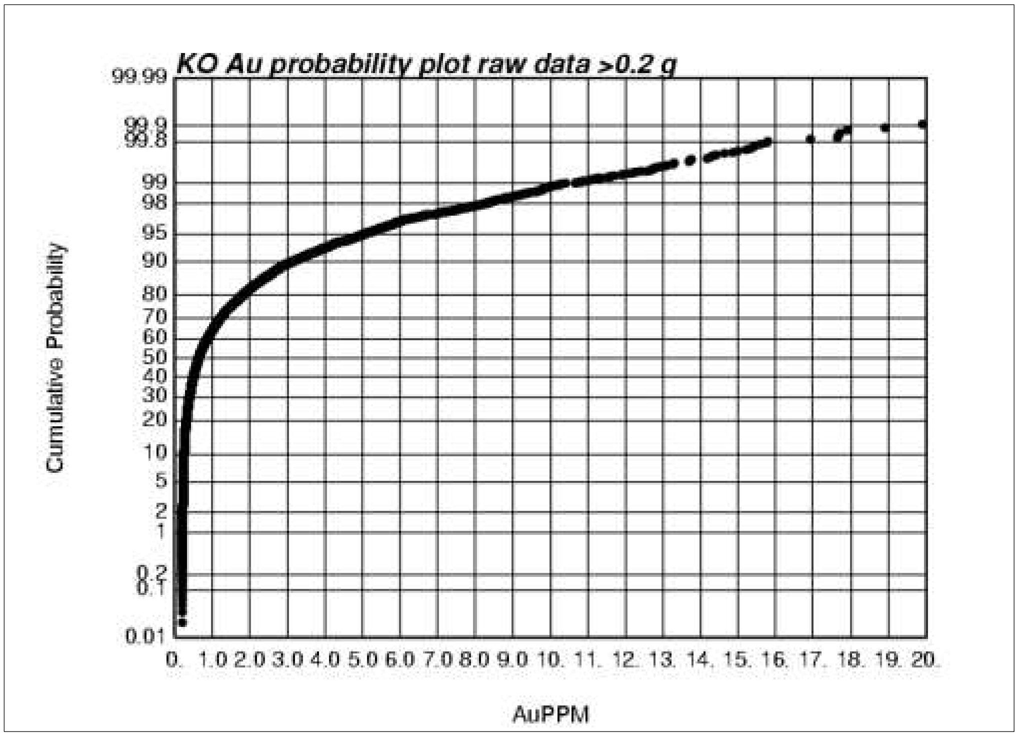
14.1.5 Capping of Assays

An assessment of high-grade Au outliers within the raw sample population was conducted for each resource area using descriptive statistics, histograms, cumulative probability plots and decile-percentile worksheets. Cap values were applied to outliers prior to compositing samples. Cap values are shown in Table 14-3. An example cumulative probability plot for the KOGD resource area is shown in Figure 14–3.

Table 14-3: Gold Cap Values

Resource Area	Au Cap (g/t)
PABL	8.5
KOGD	16.0
LU	9.5

Figure 14–3: KOGD Area Raw Au Sample Cumulative Probability Plot



Source: Golden Predator (2013)

14.1.6 Compositing

Drillholes were composited at nominal 6 metre down-hole intervals honoring lithologic contacts. Thus, composites are as close to 6 metres as possible, but always end at a lithologic contact. Partial intervals less than 3 metres length were merged with neighboring intervals. The 6 metre composite length was chosen, along with the 6 metres x 6 metres x 6 metres SMU block size for consistency between Golden Predator and RMI resource models. Composites were back-marked for SDMN and LDMN using the 3D block models created previously. Statistics for the capped and composited samples are presented in Table 14-4.

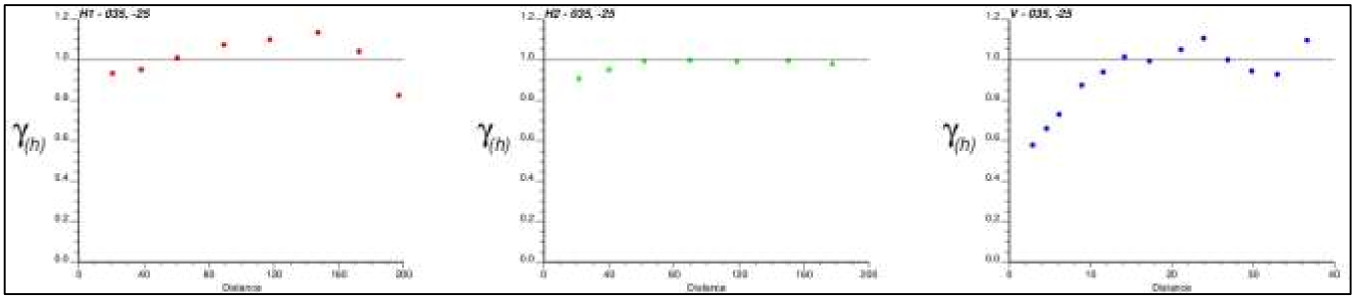
Table 14-4: Composite Gold Assay Statistics (Gold grades reported in g/t) by Zone

	Number Samples	Min	Max	Mean	Std. Dev.	Median
KOGD	9653	0.002	15.46	0.405	1.041	0.063
PABL	3616	0.000	8.50	0.450	1.006	0.077
LU	1972	0.002	7.50	0.421	0.892	0.065

14.1.7 Variography

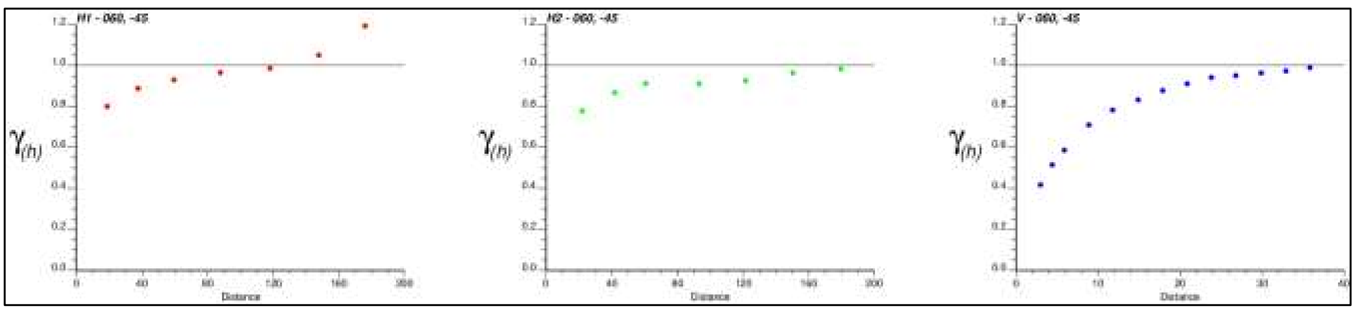
Golden Predator conducted a statistical analysis of assay data within the each Resource Area. In the Lucky area, it was determined through variography that the down-dip range of the gold grade continuity was 70 metres. The continuity along strike was 70 metres. The search ellipse ranges in the PABL and KOGD Resource Areas, 80 metres along strike and 40 metres down-dip, were determined by variography conducted previously at other, similar BRC district deposits. The variograms from the LU resource area are shown in Figure 14–4 and Figure 14–5.

Figure 14–4: Pairwise Relative Experimental Variograms within Horizontal and Vertical Directions for all Samples within SDMN1



Source: Golden Predator (2013)

Figure 14–5: Pairwise Relative Experimental Variograms within Horizontal and Vertical Directions for all Samples within SDMN2



Source: Golden Predator (2013)

14.1.8 Estimation

Within each area, blocks were estimated using only composites from the same lithologic and structural domains (KOGD and PABL only). For example, a block in structural domain 1 with a majority lithology of sedimentary will be estimated using only composites back-marked as structural domain 1 from the 3D structural model and marked as in the sediment lithology domain from the 3D lithology model. Essentially lithologic domains are hard boundaries for grade estimation. The resource was estimated in 3 passes for all blocks. A three-pass interpolation was utilized to estimate Au via an Inverse Distance method (ID) within each structural domain. Each pass searches progressively less distance; the liberal first pass fills the model with widely spaced data, much of which will contribute to the inferred category. The second pass tightly constrains the interpolation and forms the basis for much of the indicated category. The third pass constrains the interpolation to within a block or two of the composite data and assures that the blocks closest to the drillholes accurately portray the composite values. The estimation parameters are listed in Table 14-5 and Table 14-6.

Table 14-5: Block Estimation Parameters

Resource Area		1st Pass	2nd Pass	3rd Pass
KOGD	Primary Axis (metres)	80	40	3
	Secondary Axis (metres)	40	20	3
	Tertiary Axis (metres)	20	15	3

Table 14-5: Block Estimation Parameters

Resource Area		1st Pass	2nd Pass	3rd Pass
	Min # Composites	3	2	1
	Max # Composites	8	8	8
	Max Composites per Drillhole	2	2	1
	ID Power	3	3	3
PABL	Primary Axis (metres)	80	40	3
	Secondary Axis (metres)	40	20	3
	Tertiary Axis (metres)	20	15	3
	Min # Composites	3	2	1
	Max # Composites	8	8	8
	Max Composites per Drillhole	2	2	1
	ID Power	3	3	3
LU	Primary Axis (metres)	70	35	3
	Secondary Axis (metres)	70	35	3
	Tertiary Axis (metres)	18	9	3
	Min # Composites	2	3	1
	Max # Composites	8	8	8
	Max Composites per Drillhole	2	2	1
	ID Power	3	3	3

Table 14-6: Structural Domain Estimation Parameters

Resource Area	Structural Domain	Azimuth of Primary Axis	Dip of Secondary Axis
KOGD	1	110	35
	2	75	45
	3	45	55
	4	60	35
	5	45	45
	6	75	35
PABL	1	70	40
	2	65	50
	3	90	55
	4	90	55
	5	40	30
	6	50	25
	7	50	25
LU	1	35	30
	2	60	45

14.1.8.1 Estimate Validation

The model was first evaluated by comparing the composite statistics to the block model statistics for each structural domain and each lithology domain in each target area. Results are shown in Table 14-7.

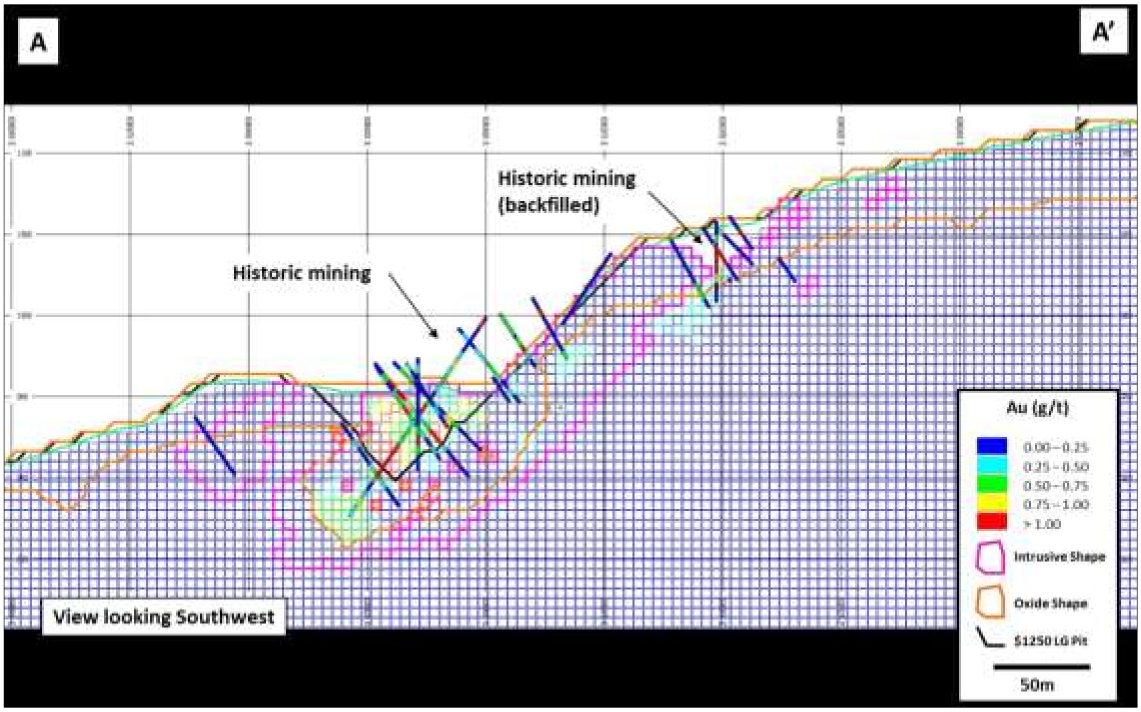
Table 14-7: Descriptive Statistics for Gold in Composite Samples and Model

Resource Area	Structural Domain	Lithology Domain	Composites				Block Model			
			Min	Max	Mean	Std. Dev.	Min	Max	Mean	Std. Dev.
KOGD	1	1	0.002	7.800	0.224	0.571	0.002	3.560	0.146	0.290
		2	0.002	4.000	0.103	0.372	0.003	1.354	0.046	0.071
	2	1	0.002	15.460	0.331	0.804	0.002	10.617	0.282	0.458
		2	0.002	3.540	0.083	0.248	0.002	3.540	0.042	0.070
	3	1	0.007	5.213	0.386	0.670	0.010	4.413	0.422	0.535
		2	0.003	3.080	0.128	0.347	0.004	1.496	0.055	0.130
	4	1	0.002	7.390	0.234	0.551	0.003	7.390	0.199	0.300
		2	0.002	2.420	0.105	0.238	0.002	2.420	0.056	0.106
	5	1	0.002	7.270	0.321	0.793	0.002	6.340	0.243	0.453
		2	0.002	4.700	0.138	0.430	0.002	2.717	0.083	0.177
	6	1	0.002	7.301	0.266	0.683	0.002	7.301	0.191	0.439
		2	0.002	11.173	0.168	0.695	0.002	3.298	0.071	0.138
PABL	1	1	0.010	1.930	0.364	0.343	0.010	1.930	0.376	0.153
		2	0.010	5.086	0.376	0.675	0.010	4.040	0.234	0.327
	2	1	0.040	2.380	0.837	0.886	0.058	2.380	0.856	0.673
		2	0.000	5.227	0.313	0.671	0.000	5.227	0.220	0.435
	3	1	0.002	1.110	0.171	0.266	0.002	1.084	0.200	0.228
		2	0.000	2.655	0.248	0.452	0.002	2.553	0.159	0.236
	4	1	0.000	3.443	0.852	1.003	0.000	3.443	0.563	0.708
		2	0.000	3.991	0.191	0.711	0.000	3.991	0.127	0.355
	5	1	0.002	1.960	0.089	0.187	0.002	1.921	0.106	0.189
		2	0.002	3.141	0.173	0.368	0.002	2.740	0.105	0.152
	6	1	0.000	7.402	0.616	0.949	0.000	7.402	0.465	0.553
		2	0.000	8.500	0.509	1.080	0.000	6.877	0.234	0.443
	7	1	0.010	0.490	0.080	0.129	0.010	0.199	0.035	0.039
		2	0.010	0.531	0.043	0.095	0.010	0.414	0.042	0.067
LU	1	1	0.010	7.500	0.681	1.080	0.010	7.057	0.487	0.627
		2	0.005	5.210	0.169	0.473	0.006	5.013	0.071	0.208
	2	1	0.002	7.500	0.493	0.997	0.002	6.243	0.339	0.573
		2	0.002	4.930	0.112	0.446	0.002	4.930	0.068	0.175

(Gold grade reported in g/t)

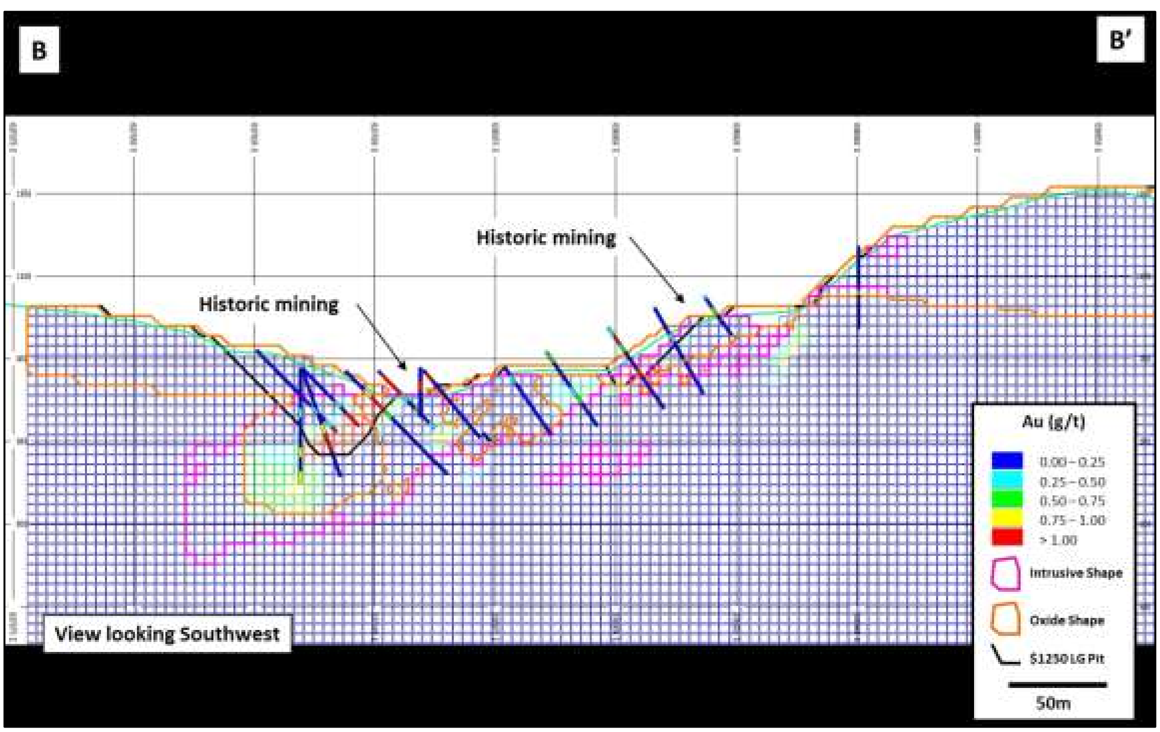
The model was validated by evaluating the blocks against actual composite assay data to determine if the estimated blocks fit the grade and parameters of the deposit. A cross section for each resource area displaying the block model gold content with the composite gold data is presented in Figure 14–6, Figure 14–7, Figure 14–9, Figure 14–10, and Figure 14–12. Locations of drillholes and sections given in Figure 14–8, Figure 14–11, and Figure 14–13.

Figure 14–6: Validation Section A-A' of Kokanee Block Model (See Figure 14–8 for Location)



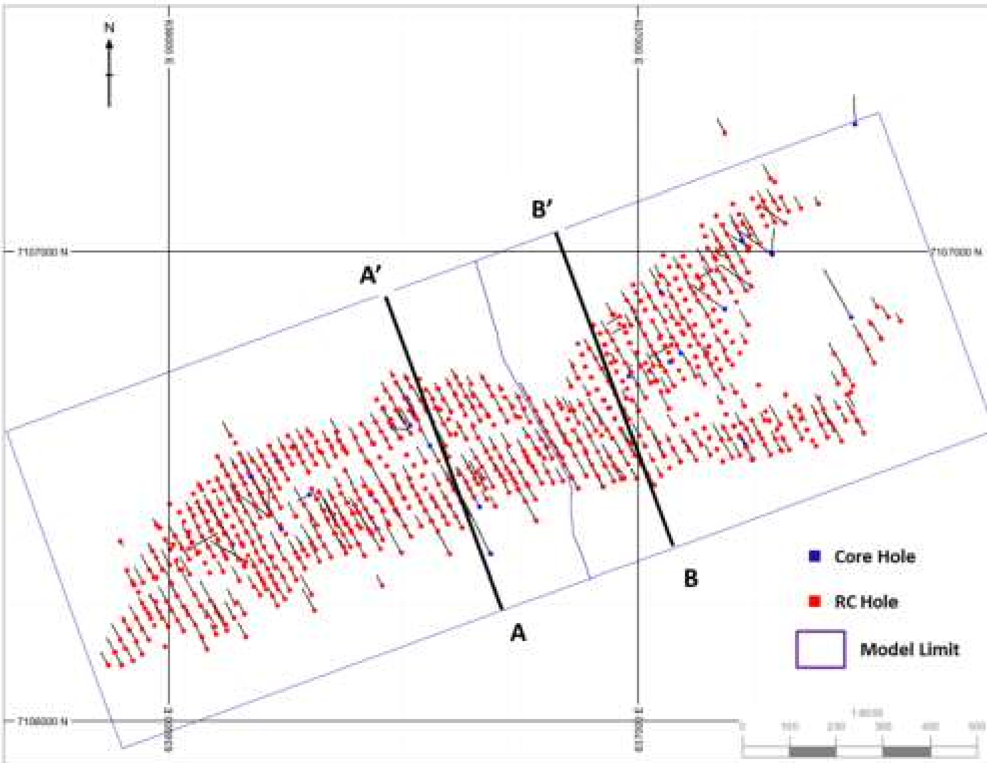
Source: Golden Predator (2013)

Figure 14–7: Validation Section B-B' of Golden Block Model (See Figure 14–8 for Location)



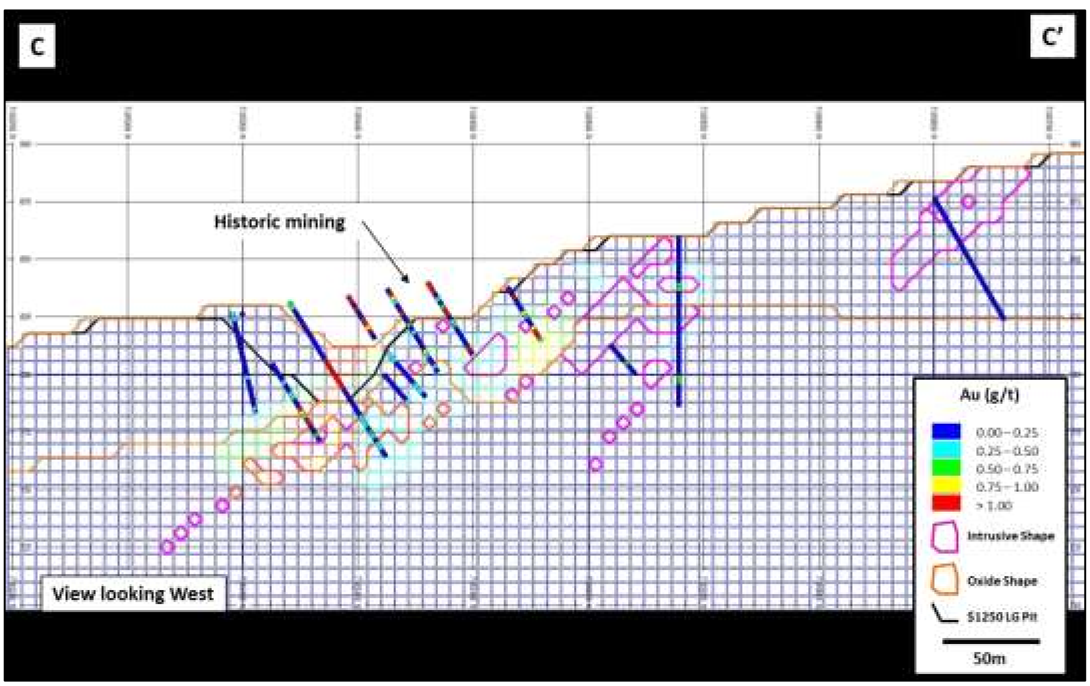
Source: Golden Predator (2013)

Figure 14–8: Detailed Drillhole Locations of KOGD and Location of Cross Sections A-A' and B-B'



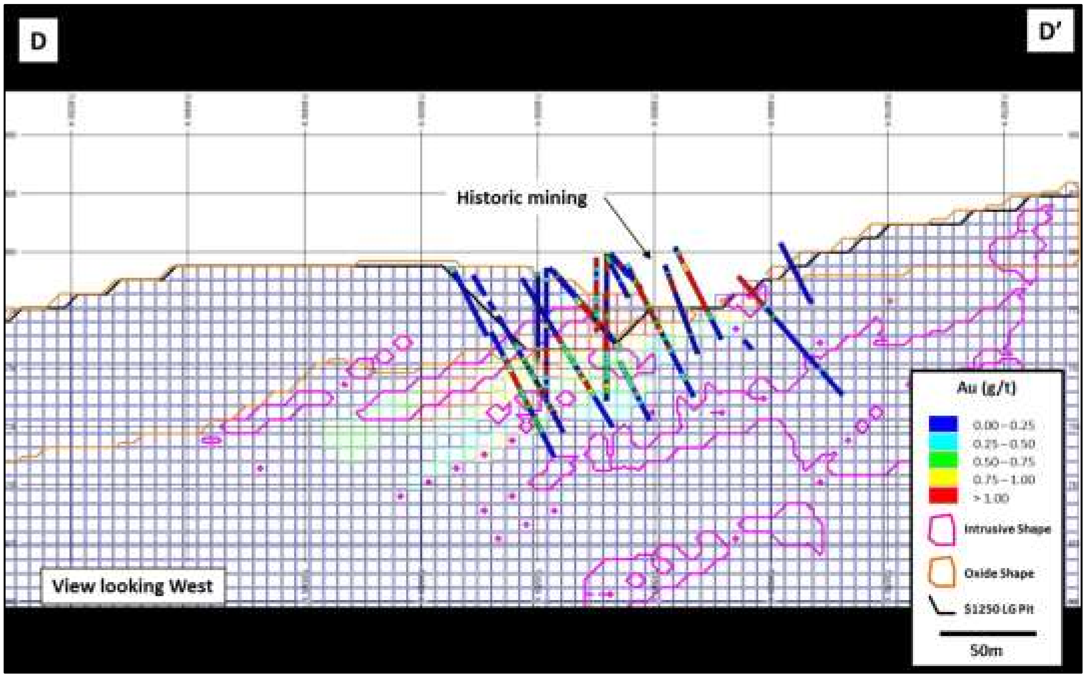
Source: Golden Predator (2013)

Figure 14–9: Validation Section C-C' of Pacific Block Model (See Figure 14–11 for Location)



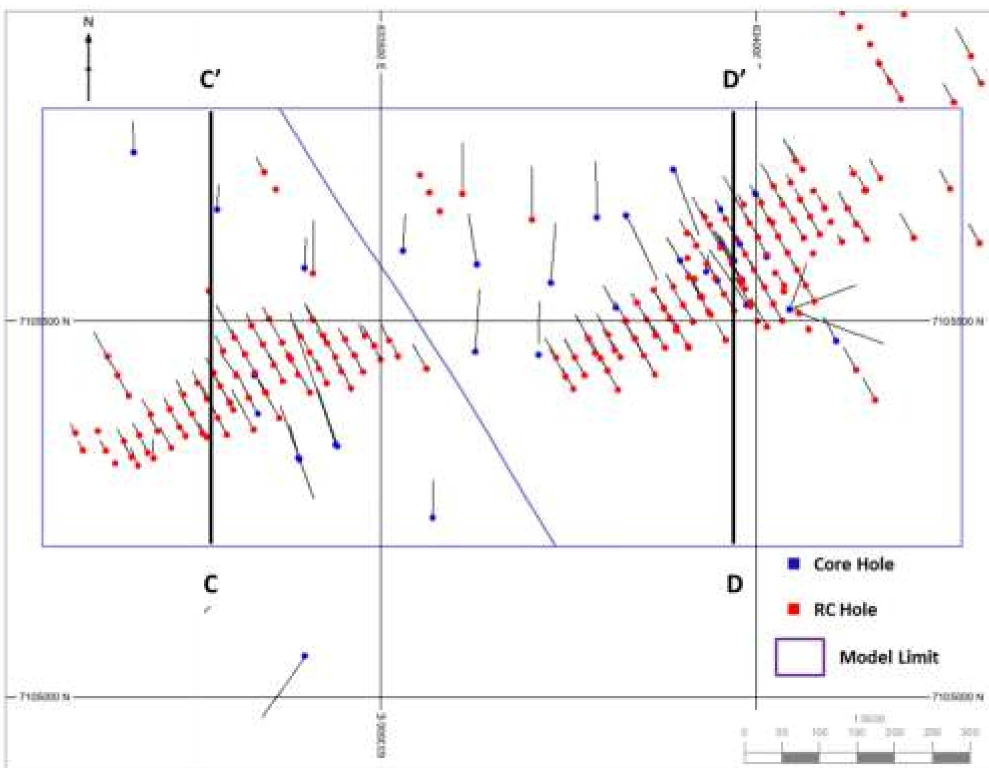
Source: Golden Predator (2013)

Figure 14–10: Validation Section D-D' of BL Block Model (See Figure 14–11 for Location)



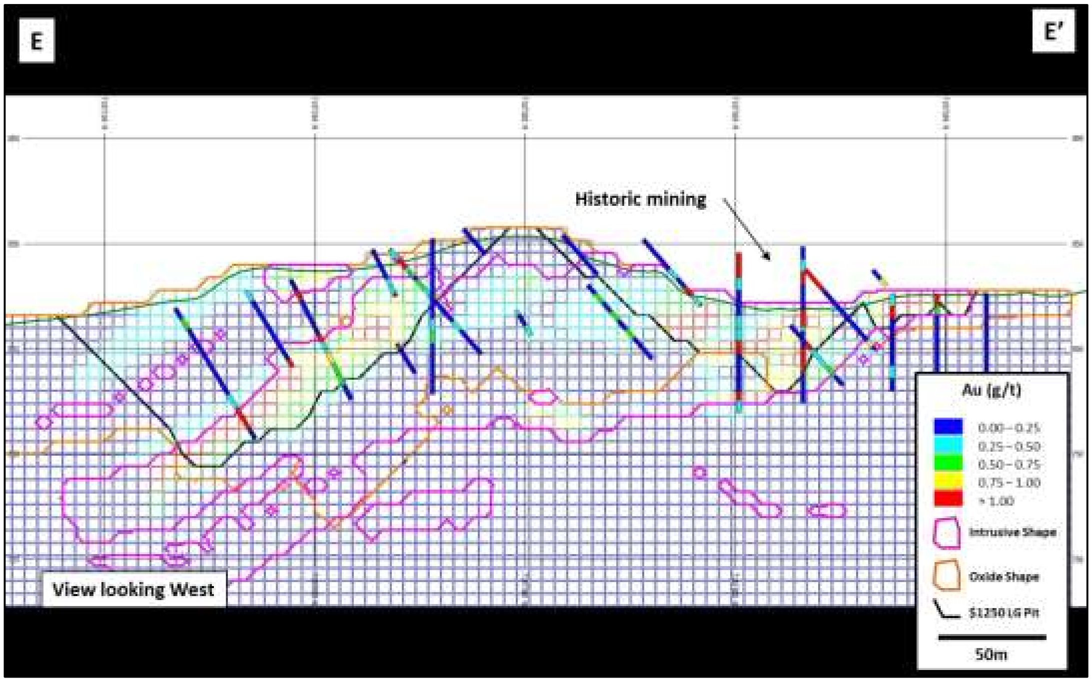
Source: Golden Predator (2013)

Figure 14–11: Detailed Drillhole Locations of PABL and Location of Cross Sections C-C' and D-D'



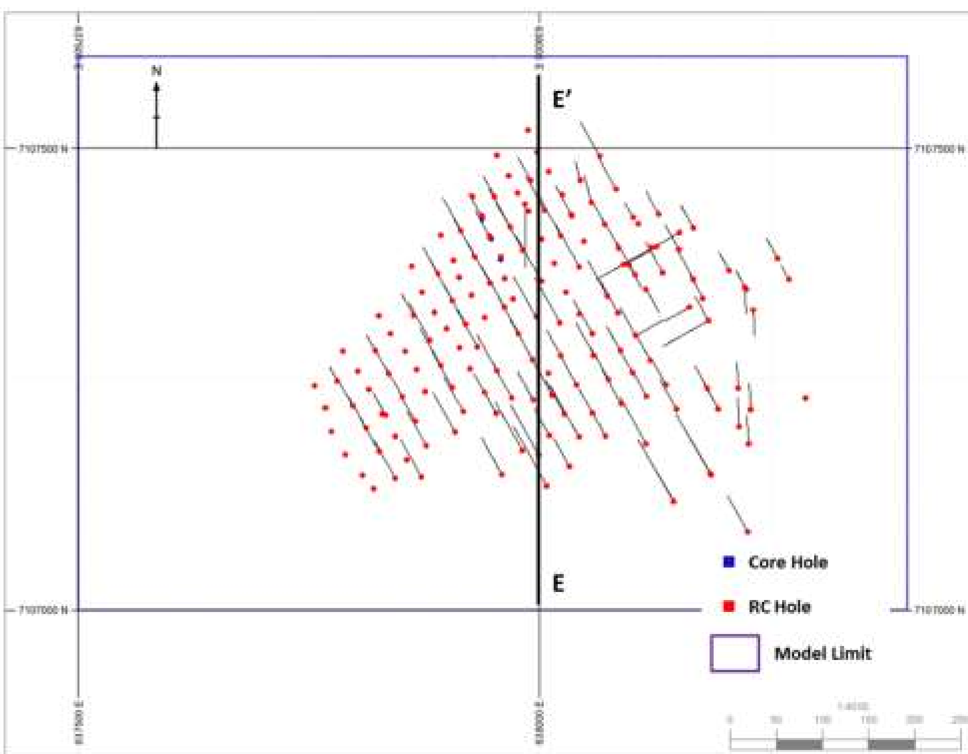
Source: Golden Predator (2013)

Figure 14–12: Validation Section E-E' of Lucky Block Model (See Figure 14–13 for Location)



Source: Golden Predator (2013)

Figure 14–13: Detailed Drillhole Locations of LU and Location of Cross Section E-E



Source: Golden Predator (2013)

The resources were also evaluated using a nearest neighbor (NN) estimation as a check. NN estimations are valid at a 0 g/t cut-off only; the 0 g/t cut-off is shown for illustrative purposes only. The resource estimate produced using this method, as well as the percent change between the ID3 and NN methods are shown in Table 14-8.

Table 14-8: ID3 and NN Model Comparison

Resource Area	ID Avg. Grade 0.0 cut-off (g/t)	NN Avg. Grade 0.0 cut-off (g/t)	% Difference
PABL	0.186	0.195	-4.7%
KOGD	0.163	0.160	1.8%
LU	0.228	0.228	0.2%

14.1.9 Review of Estimate – Gustavson

Gustavson reviewed the estimates prepared by Golden Predator through a series of comparisons. The estimates were created using the exploration drill data, but not the blast-hole data from the previous operation. Gustavson then performed a series of tests including:

- Visual comparison of the exploration data to the model on plans and sections.
- Within the shell of the previously mined pit:
  - Visual comparison of the exploration data to the Blast data on plans and sections.
  - Visual comparison of the blast data to the model on plans and sections.
  - A comparison of Golden Predator variograms and modeling parameters with the variography of the blast-hole data.

Gustavson also compared the statistics (cumulative frequency distributions) of the exploration data, composites used for the estimate and the model as well as with the blast data inside of the mined pit shells. This allowed for the comparison of the mean grades within the data as well as comparison of the expected volume variance change between the composites and the estimated blocks.

Gustavson concluded that the modeling complied with CIM Best Practice, and is appropriate for reporting mineral resources.

14.2 Bohemian, Schooner, Lower Fosters, West Big Rock, East Big Rock, Classic, and Lone Star Deposits

Mr. Michael J. Lechner, P. Geo., President of RMI has completed updated Mineral Resources for the Bohemian (BH), Schooner (SC), Lower Fosters (FS), West Big Rock (WB), East Big Rock (EB) (formerly known as Big Rocks) and Classic deposits. In addition, Mr. Lechner has completed a new Mineral Resource estimate for the Lone Star deposit. The Classic and Lonestar deposits were not included as part of the PEA due to their location and lack of detailed as their mineral resource estimates did not meet the current objectives of Golden Predator.

RMI worked closely with Golden Predator's geologic staff in preparing geologic models for each of the deposits. Most of the modeling was completed using MineSight® software. Various statistical analyses were completed using proprietary software.

14.2.1 Deposit Geology Pertinent to Resource Estimation

The deposits at the Brewery Creek Project are primarily hosted by altered quartz monzonite sills, which commonly contain bifurcating and disconnected lenses of sedimentary strata. The sedimentary lenses are often thin and difficult to model using standard wire framing techniques due to limited hole-to-hole continuity and variable thicknesses of the intercalated sills and sedimentary rocks. Due to these constraints, a probability technique was used to predict the distribution of intrusive and sedimentary rocks in all deposits except Classic and Lone Star.

Raw lithologic data collected from drill logs were coded with an integer value of 1 if a specific lithologic unit was present, a 0 was entered if that lithology was not present in the drillhole interval. A total of six lithologic units were modeled using this probability method. Block models were constructed using 2m x 2m x 2m blocks and 2-metre-long drillhole composites. A two pass inverse distance squared estimation method was used to estimate probabilities for the six unique lithologic units. After all of the lithologic indicators were estimated the blocks were assigned intrusive (1) and sediment (2) codes based on the highest of the six possible lithologic probabilities that were estimated. The 2-metre model blocks provided a high degree of resolution and resulted in excellent continuity for the distribution of thin sedimentary intervals. Three dimensional intrusive wireframes were constructed from the 2-metre block model. Those wireframes were used to code the 6-metre block models so that the blocks were either intrusive or sediment based on a majority rule. In addition to whole block coding, the percentage of intrusive and sediment were stored in each block. This step would allow mine planners to have an idea about possible dilution of "clean" intrusive with potentially preg robbing sedimentary rocks.

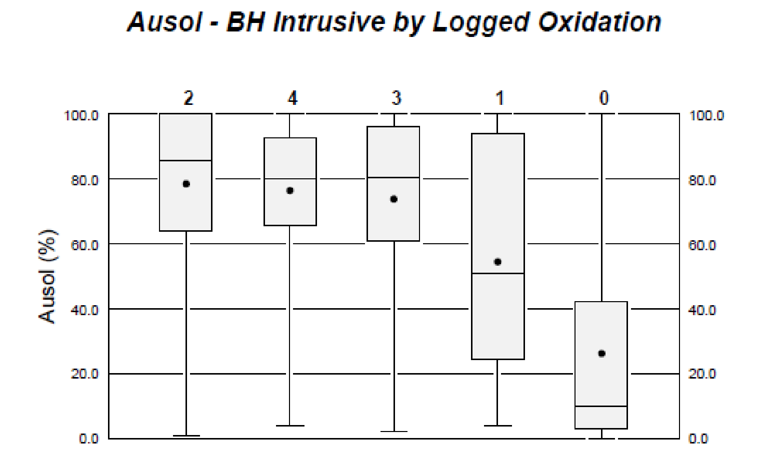
Wide-spaced drilling and complicated compositional and textural relationships in the Classic-Lone Star intrusive complex precluded the construction of a standard wire frame geological model. Probability modeling of lithology as described above was deemed to not be appropriate for the style of mineralization observed in this area, so development of a technique suited to the construction of a model was required for this intrusive suite.

A multi-variate factor analysis of ICP data from the Classic and Lone Star resource drilling resulted in three whole-rock signatures and one hydrothermal signature. This analysis, based on empirical geochemical data, was used to build a model which is thought to be more appropriate than from drillhole lithologic data alone. The factor representing a hydrothermal signature of gold, arsenic, and copper defined a broad and coherent area of mineralization. Results of the analysis were used to construct a three dimensional wireframe that was used to constrain the estimate of gold for the Classic-Lone Star deposits.

14.2.1.1 Oxidation

Previously, Golden Predator constructed oxidation surfaces based on logged oxidation intensity. Numeric codes of 0 (no oxidation) through 4 (complete oxidation) were assigned to each interval by the Golden Predator geologic staff. The oxide surface was based on intervals where the oxide intensity code was 3 or 4. Cyanide soluble analyses were collected for 2012 drillhole samples where the initial fire assay grade was above a 0.2 g/t cut-off. RMI conducted a study of comparing how well solubility data compared with the logged oxidation attribute. In general, high cyanide solubility results tended to correspond well with intervals that were logged with oxidation codes of 3 and 4 for intrusive rocks. However, RMI found that in many cases, reasonably high solubility (+ 70%) were associated with intervals that had been logged with oxidation codes of 1 or 2. In general, gold solubility was found to be quite low for sedimentary units. In addition to poor gold solubility the sedimentary rocks also display varying degrees of gold preg robbing. Boxplots were generated for each mineralized zone by major rock type (intrusive and sedimentary rocks) comparing cyanide solubility with logged oxidation codes. Figure 14–14 shows such a boxplot which shows various gold solubility statistics for each logged oxidation intensity codes. The data in Figure 14–14 show that there is a reasonably high solubility ratio for logged oxidation codes 2, 3, and 4.

Figure 14–14: Au Solubility vs. Logged Oxidation (BH Intrusives)

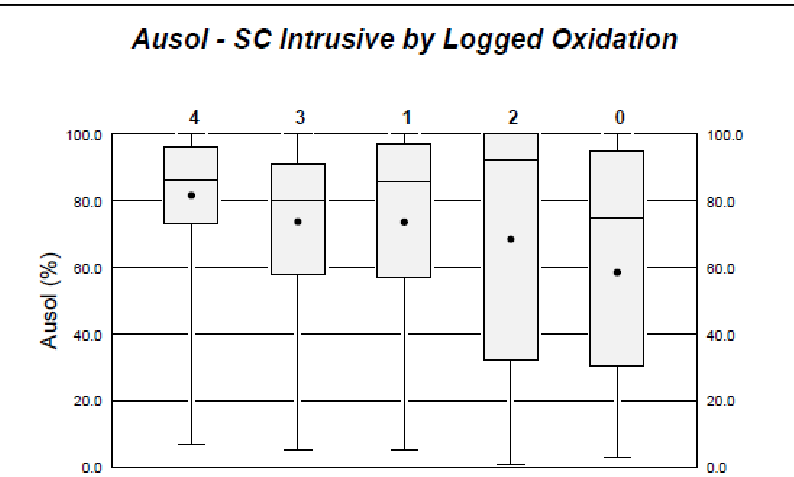


	2	4	3	1	0	
Number of data	73	114	84	23	118	Number of data
Mean	78.463	76.4	73.737	54.466	26.193	Mean
Maximum	100.0	100.0	100.0	100.0	100.0	Maximum
Upper quartile	100.0	92.8	96.0	93.893	42.0	Upper quartile
Median	85.523	80.0	80.5	50.853	10.0	Median
Lower quartile	63.723	65.616	60.685	24.475	3.0	Lower quartile
Minimum	1.0	4.0	2.0	4.0	0.0	Minimum
Variance	576.074	466.187	699.844	1095.4	982.234	Variance
CV	0.306	0.283	0.359	0.608	1.197	CV
Skewness	-1.141	-1.158	-1.065	0.152	1.303	Skewness

Source: RMI (2013)

Figure 14–15 shows a similar boxplot of gold solubility of Schooner intrusives and in this case even logged oxidation code 1 shows a reasonably high solubility ratio.

Figure 14–15: Au Solubility vs. Logged Oxidation (SC Intrusives)



	4	3	1	2	0	
Number of data	153	151	70	87	3	Number of data
Mean	81.584	73.655	73.57	68.421	58.461	Mean
Maximum	100.0	100.0	100.0	100.0	100.0	Maximum
Upper quartile	96.0	91.0	97.0	100.0	94.938	Upper quartile
Median	86.0	80.0	85.622	92.165	74.75	Median
Lower quartile	73.0	58.0	57.0	32.0	30.482	Lower quartile
Minimum	7.0	5.0	5.0	1.0	3.0	Minimum
Variance	353.164	470.859	744.267	1365.44	2007.32	Variance
CV	0.23	0.295	0.371	0.54	0.766	CV
Skewness	-1.484	-0.778	-0.995	-0.688	-0.316	Skewness

Source: RMI (2013)

Based on an analysis of solubility data a new method was used to sub-divide the various block models into two regimes: potentially leachable ("oxidized") and non-leachable ("sulfide"). An oxidation indicator probability function was implemented to help predict potentially leachable material. Indicator values of 0 (low probability of being leachable) or 1 (potentially leachable) were assigned to six-metre-long drillhole composites. The indicator value of 1 was based on the previously described boxplot analysis on a deposit by deposit basis. The 0/1 indicator values were used to estimate "oxidation" probability in each block model. Blocks with an estimated "oxidation" probability above 50% were flagged as potentially amenable to cyanidation (i.e. MODOX = 1). All other blocks were considered as "sulfide" (i.e. MODOX = 2). The MODOX field was then used to define oxide and sulfide resources.

14.2.2 Data Used for Estimation

RMI was provided with a series of Excel spreadsheets that contained collar, survey, assay, geologic, and metallurgical data. The records from these files were imported into MineSight® and used to estimate Mineral Resources after a number of statistical studies were completed. Table 14-9 summarizes the type, number, and metres of drilling data for each mineralized area that was used by Mr. Lechner.

Table 14-9: Drillhole Data

Resource Area	Core		RC		Total	
	Count	Metres	Count	Metres	Count	Metres
Bohemian	41	4,673	113	8,642	154	13,315
Schooner	81	8,394	16	1,676	97	10,070
Lower Fosters	40	3,250	396	16,388	436	19,638
West Big Rock	60	6,209	100	7,648	160	13,857
East Big Rock	17	1,925	114	7,461	131	9,386
Classic	17	4,088	52	9,391	69	13,478
Lone Star	17	3,865	12	2,283	29	6,147
Total	273	32,403	803	53,489	1,076	85,892

14.2.2.1 Drillhole Assay Statistics

Drillhole assay statistics were generated for fire assay gold (AuFA), cyanide soluble (AuCN), and gold preg rob. The statistics were tabulated by mineralized area, sample type, major rock type, and logged oxidation code. The statistics are summarized in Table 14-10 through Table 14-15. The statistics (drilled metreage, mean grade, incremental metreage above cut-off, grade-thickness products, incremental grade-thickness, standard deviation, and coefficient of variation are shown for four cut-off grades.

Table 14-10: Drillhole Assay Statistics – Gold Fire Assays by Area

Resource Area	Uncapped AuFA Statistics Above Cut-off							
	AuFA Cut-off	Total Metres	Inc. Percent	Mean AuFA (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0	68,440	78%	0.28	18,879	11.50%	0.98	3.55
	0.2	14,872	11%	1.12	16,713	11.90%	1.87	1.66
	0.5	7,630	3%	1.9	14,464	5.90%	2.36	1.25
	0.7	5,740	8%	2.33	13,354	70.70%	2.58	1.11

Table 14-10: Drillhole Assay Statistics – Gold Fire Assays by Area

Resource Area	Uncapped AuFA Statistics Above Cut-off							
	AuFA Cut-off	Total Metres	Inc. Percent	Mean AuFA (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
Bohemian	0	12,106	72%	0.42	5,074	8.50%	1.4	3.35
	0.2	3,371	13%	1.38	4,643	9.60%	2.41	1.75
	0.5	1,804	3%	2.3	4,155	4.20%	3	1.3
	0.7	1,443	12%	2.73	3,943	77.70%	3.21	1.18
Schooner	0	9,034	78%	0.45	4,023	5.30%	1.61	3.62
	0.2	2,009	9%	1.9	3,812	6.00%	2.99	1.58
	0.5	1,231	3%	2.9	3,569	3.50%	3.47	1.2
	0.7	987	11%	3.47	3,428	85.20%	3.65	1.05
Lower Fosters	0	5,928	76%	0.36	2,113	6.90%	1.09	3.04
	0.2	1,439	8%	1.37	1,966	7.20%	1.87	1.37
	0.5	952	3%	1.91	1,815	5.30%	2.1	1.1
	0.7	761	13%	2.24	1,704	80.60%	2.23	1
West Big Rock	0	13,017	80%	0.23	2,938	9.90%	0.65	2.89
	0.2	2,568	8%	1.03	2,648	11.90%	1.16	1.12
	0.5	1,478	3%	1.56	2,300	6.80%	1.29	0.83
	0.7	1,140	9%	1.84	2,100	71.50%	1.34	0.73
East Big Rock	0	9,100	82%	0.2	1,811	13.80%	0.52	2.62
	0.2	1,662	8%	0.94	1,561	12.20%	0.9	0.96
	0.5	974	3%	1.38	1,340	8.20%	0.96	0.7
	0.7	722	8%	1.65	1,192	65.80%	0.97	0.59
Classic	0	13,190	78%	0.16	2,083	29.40%	0.31	1.98
	0.2	2,925	16%	0.5	1,470	30.20%	0.53	1.05
	0.5	849	3%	0.99	842	10.30%	0.78	0.78
	0.7	478	4%	1.31	627	30.10%	0.92	0.7
Lone Star	0	6,065	85%	0.14	835	26.80%	0.44	3.18
	0.2	898	9%	0.68	611	20.10%	0.97	1.43
	0.5	344	2%	1.29	443	9.70%	1.36	1.05
	0.7	208	3%	1.74	362	43.30%	1.59	0.92

Table 14-11: Drillhole Assay Statistics – Gold Fire Assays by Sample Type

Type	Uncapped AuFA Statistics Above Cut-off							
	AuFA Cut-off	Total Metres	Inc. Percent	Mean AuFA (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0	68,440	78%	0.28	18,879	11.50%	0.98	3.55
	0.2	14,872	11%	1.12	16,713	11.90%	1.87	1.66
	0.5	7,630	3%	1.9	14,464	5.90%	2.36	1.25
	0.7	5,740	8%	2.33	13,354	70.70%	2.58	1.11
Core	0	27,425	80%	0.32	8,769	8.40%	1.25	3.9
	0.2	5,389	9%	1.49	8,031	8.40%	2.49	1.67

	0.5	3,008	2%	2.42	7,290	4.30%	3.03	1.25
	0.7	2,363	9%	2.92	6,912	78.80%	3.24	1.11
RC	0	41,016	77%	0.25	10,110	14.10%	0.75	3.03
	0.2	9,483	12%	0.92	8,682	14.90%	1.35	1.47
	0.5	4,622	3%	1.55	7,175	7.20%	1.71	1.1
	0.7	3,376	8%	1.91	6,442	63.70%	1.89	0.99

Table 14-12: Drillhole Assay Statistics – Gold Fire Assays by Major Rock Type

Type	Uncapped AuFA Statistics Above Cut-off							
	AuFA Cut-off	Total Metres	Inc. Percent	Mean AuFA (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0	68,440	78%	0.28	18,879	11.50%	0.98	3.55
	0.2	14,872	11%	1.12	16,713	11.90%	1.87	1.66
	0.5	7,630	3%	1.9	14,464	5.90%	2.36	1.25
	0.7	5,740	8%	2.33	13,354	70.70%	2.58	1.11
Undefined	0	186	88%	0.12	23	27.90%	0.32	2.55
	0.2	23	7%	0.72	17	15.80%	0.62	0.85
	0.5	11	2%	1.21	13	7.40%	0.6	0.5
	0.7	8	4%	1.47	11	48.80%	0.52	0.35
Intrusive	0	41,544	70%	0.39	16,188	9.00%	1.2	3.08
	0.2	12,336	14%	1.19	14,725	11.20%	1.98	1.66
	0.5	6,536	4%	1.98	12,918	5.60%	2.47	1.25
	0.7	4,979	12%	2.41	12,003	74.10%	2.69	1.12
Sediment	0	26,440	91%	0.1	2,641	25.90%	0.4	4.04
	0.2	2,479	5%	0.79	1,956	16.30%	1.09	1.39
	0.5	1,076	1%	1.42	1,526	7.30%	1.43	1.01
	0.7	748	3%	1.78	1,333	50.50%	1.59	0.89
Overburden	0	271	87%	0.1	27	42.40%	0.2	1.97
	0.2	34	10%	0.45	15	27.80%	0.38	0.85
	0.5	8	1%	0.98	8	7.10%	0.48	0.49
	0.7	5	2%	1.2	6	22.60%	0.49	0.41

Table 14-13: Drillhole Assay Statistics – Gold Fire Assays by Logged Oxidation

Type	Uncapped AuFA Statistics Above Cut-off							
	AuFA Cut-off	Total Metres	Inc. Percent	Mean AuFA (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0	68,440	78%	0.28	18,879	11.50%	0.98	3.55
	0.2	14,872	11%	1.12	16,713	11.90%	1.87	1.66
	0.5	7,630	3%	1.9	14,464	5.90%	2.36	1.25
	0.7	5,740	8%	2.33	13,354	70.70%	2.58	1.11
Undefined	0	13,000	78%	0.24	3,123	14.50%	0.89	3.72
	0.2	2,889	12%	0.92	2,670	15.60%	1.73	1.87
	0.5	1,313	3%	1.66	2,183	7.10%	2.36	1.42

Table 14-13: Drillhole Assay Statistics – Gold Fire Assays by Logged Oxidation

Type	Uncapped AuFA Statistics Above Cut-off							
	AuFA Cut-off	Total Metres	Inc. Percent	Mean AuFA (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
	0.7	936	7%	2.1	1,962	62.80%	2.68	1.28
0	0	13,792	87%	0.17	2,299	16.00%	0.71	4.26
	0.2	1,741	6%	1.11	1,930	11.50%	1.72	1.56
	0.5	892	2%	1.87	1,666	5.70%	2.15	1.15
	0.7	669	5%	2.29	1,535	66.80%	2.33	1.02
1	0	12,041	84%	0.18	2,142	17.80%	0.7	3.93
	0.2	1,922	9%	0.92	1,761	16.00%	1.55	1.69
	0.5	798	2%	1.78	1,418	6.20%	2.12	1.2
	0.7	569	5%	2.26	1,285	60.00%	2.35	1.04
2	0	9,616	77%	0.29	2,778	11.80%	1.12	3.88
	0.2	2,174	12%	1.13	2,450	12.80%	2.16	1.91
	0.5	1,025	3%	2.05	2,096	5.70%	2.87	1.41
	0.7	755	8%	2.57	1,937	69.70%	3.19	1.24
3	0	10,533	72%	0.35	3,651	8.30%	0.93	2.68
	0.2	2,998	12%	1.12	3,349	11.00%	1.48	1.32
	0.5	1,715	4%	1.72	2,949	6.50%	1.72	1
	0.7	1,313	12%	2.07	2,713	74.30%	1.83	0.89
4	0	9,459	67%	0.52	4,886	6.80%	1.47	2.84
	0.2	3,148	13%	1.45	4,552	8.20%	2.27	1.57
	0.5	1,889	4%	2.2	4,153	4.70%	2.68	1.22
	0.7	1,497	16%	2.62	3,922	80.30%	2.87	1.1

Table 14-14: Drillhole Assay Statistics – Cyanide Soluble Gold Assays by Area

Resource Area	Uncapped AuCN Statistics Above Cut-off							
	AuCN Cut-off	Total Metres	Inc. Percent	Mean AuCN (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0	6,688	40%	0.56	3,738	5.90%	1.06	1.89
	0.2	3,995	32%	0.88	3,517	18.00%	1.27	1.44
	0.5	1,829	7%	1.55	2,842	7.70%	1.64	1.05
	0.7	1,339	20%	1.91	2,555	68.40%	1.79	0.94
Bohemian	0	665	57%	0.63	420	4.90%	1.34	2.13
	0.2	284	14%	1.41	399	7.00%	1.78	1.27
	0.5	191	6%	1.94	370	5.40%	1.96	1.01
	0.7	153	23%	2.26	347	82.70%	2.06	0.91
Schooner	0	859	39%	1.14	977	2.30%	2.09	1.84
	0.2	520	22%	1.83	954	6.00%	2.45	1.34
	0.5	333	4%	2.69	896	2.20%	2.71	1.01
	0.7	297	35%	2.95	874	89.50%	2.76	0.94

Table 14-14: Drillhole Assay Statistics – Cyanide Soluble Gold Assays by Area

Resource Area	Uncapped AuCN Statistics Above Cut-off							
	AuCN Cut-off	Total Metres	Inc. Percent	Mean AuCN (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
Lower Fosters	0	232	44%	0.48	112	4.10%	0.69	1.43
	0.2	130	26%	0.83	107	16.40%	0.76	0.92
	0.5	70	8%	1.26	89	9.30%	0.8	0.63
	0.7	52	22%	1.51	78	70.20%	0.79	0.52
West Big Rock	0	1,407	43%	0.52	726	5.10%	0.77	1.49
	0.2	806	26%	0.85	689	16.20%	0.87	1.02
	0.5	446	9%	1.28	571	9.90%	0.98	0.76
	0.7	323	23%	1.54	499	68.70%	1.03	0.67
East Big Rock	0	480	31%	0.74	357	3.30%	0.85	1.14
	0.2	333	23%	1.04	345	9.80%	0.87	0.84
	0.5	222	9%	1.4	310	7.50%	0.85	0.61
	0.7	177	37%	1.6	284	79.40%	0.84	0.52
Classic	0	2,280	42%	0.33	758	13.40%	0.42	1.27
	0.2	1,326	42%	0.5	657	38.10%	0.49	0.99
	0.5	378	7%	0.97	368	11.90%	0.71	0.73
	0.7	222	10%	1.25	277	36.60%	0.82	0.65
Lone Star	0	766	22%	0.51	390	6.10%	0.73	1.43
	0.2	596	53%	0.61	366	32.50%	0.79	1.29
	0.5	190	10%	1.26	239	11.20%	1.16	0.93
	0.7	115	15%	1.7	196	50.20%	1.32	0.78

Table 14-15 : Drillhole Assay Statistics – Gold Preg Rob Assays by Area

Resource Area	Uncapped Preg Rob Statistics Above Cut-off							
	Preg Rob Cut-off	Total Metres	Inc. Percent	Mean Preg Rob (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0%	4,392	87%	12%	506	14.20%	0.27	2.34
	25%	563	3%	77%	434	9.90%	0.25	0.33
	50%	428	2%	90%	384	8.60%	0.13	0.14
	75%	361	8%	94%	341	67.40%	0.07	0.08
Bohemian	0%	658	88%	12%	81	12.80%	0.29	2.37
	25%	80	1%	88%	70	2.50%	0.19	0.22
	50%	74	1%	93%	68	5.60%	0.11	0.12
	75%	67	10%	96%	64	79.10%	0.06	0.07
Schooner	0%	729	84%	14%	103	12.30%	0.3	2.14
	25%	115	5%	79%	91	11.80%	0.28	0.35
	50%	82	0%	96%	78	1.00%	0.07	0.07
	75%	80	11%	96%	77	74.90%	0.06	0.06

Table 14-15 : Drillhole Assay Statistics – Gold Preg Rob Assays by Area

Resource Area	Uncapped Preg Rob Statistics Above Cut-off							
	Preg Rob Cut-off	Total Metres	Inc. Percent	Mean Preg Rob (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
Lower Fosters	0%	66	51%	42%	28	9.90%	0.38	0.9
	25%	32	10%	77%	25	8.90%	0.21	0.27
	50%	26	2%	87%	22	4.10%	0.1	0.11
	75%	24	37%	87%	21	77.10%	0.1	0.11
West Big Rock	0%	1,139	72%	24%	270	11.20%	0.35	1.48
	25%	316	6%	76%	239	9.90%	0.25	0.33
	50%	244	5%	87%	213	13.00%	0.14	0.16
	75%	189	17%	94%	178	65.90%	0.07	0.08
East Big Rock	0%	84	83%	11%	9	36.20%	0.16	1.46
	25%	15	14%	40%	6	44.30%	0.13	0.33
	50%	3	3%	64%	2	19.40%	0.05	0.08
	75%	0	0%	0%	0	0.00%	0	0
Classic	0%	1,718	100%	1%	16	78.70%	0.04	3.9
	25%	7	0%	52%	3	16.40%	0.11	0.21
	50%	1	0%	78%	1	0.00%	0	0
	75%	1	0%	78%	1	4.90%	0	0

14.2.2.2 Topographic Data

RMI was provided with three dimensional topographic surfaces that were created by Golden Predator based on a LiDAR survey that was conducted in 2012. These surfaces were used to determine the percentage of rock in each model block.

14.2.3 Bulk Density

Bulk density determinations were performed by Golden Predator on drill core samples during their 2011 and 2012 drilling campaigns. A total of 851 bulk density determinations were collected from the Bohemian, Schooner, Lower Fosters, West Big Rock and East Big Rock zones. The determinations were made by weighing select core samples in air and water using a triple beam balance. The bulk density determinations were examined by a variety of logged attributes. RMI ultimately elected to differentiate density based on rock type (intrusive or sediment) and oxidation (oxide or sulfide). Table 14-16 summarizes the bulk density data that were used for the Bohemian, Schooner, Lower Fosters, West Big Rock and East Big Rock block models.

Table 14-16: Bulk Density for BH, SC, FS, WB, and EB Models

Major Rock Type – Oxidation	Count	Mean SG (g/cm^3)	Density of Models (g/cm^3)
Intrusive Oxide	265	2.57	2.57
Intrusive Sulfide	125	2.64	2.64
All Intrusives	390	2.59	n/a
Sediment Oxide	4	2.59	2.57
Sediment Sulfide	67	2.67	2.67

Table 14-16: Bulk Density for BH, SC, FS, WB, and EB Models

Major Rock Type – Oxidation	Count	Mean SG (g/cm³)	Density of Models (g/cm³)
All Sedimentary rocks	461	2.67	n/a

An additional 111 bulk density samples were collected from the Classic and Lone Star deposits. Based on an analysis of that data, RMI chose to use a single bulk density value of 2.73 g/cm³ for the Classic and Lone Star deposit models.

14.2.4 Methodology

Four MineSight® block models were constructed by RMI for estimating Mineral Resources for seven distinct zones. A block size of 6m x 6m x 6m was selected for all models because this dimension is thought to represent a reasonable selective mining unit (SMU). Three of the block models were not rotated and their areal extents are summarized in Table 14-17.

Table 14-17: Block Model Extents

Resource Area	Easting			Northing			Elevation		
	Min	Max	No. Cols.	Min	Max	No. Rows	Min	Max	No. Levels
Bohemian & Schooner	638,322	639,432	185	7,106,887	7,107,379	82	697	937	40
Lower Fosters	635,304	635,832	88	7,105,754	7,106,204	75	646	958	52
West & East Big Rock	630,797	632,507	285	7,105,547	7,106,153	10	597	855	43

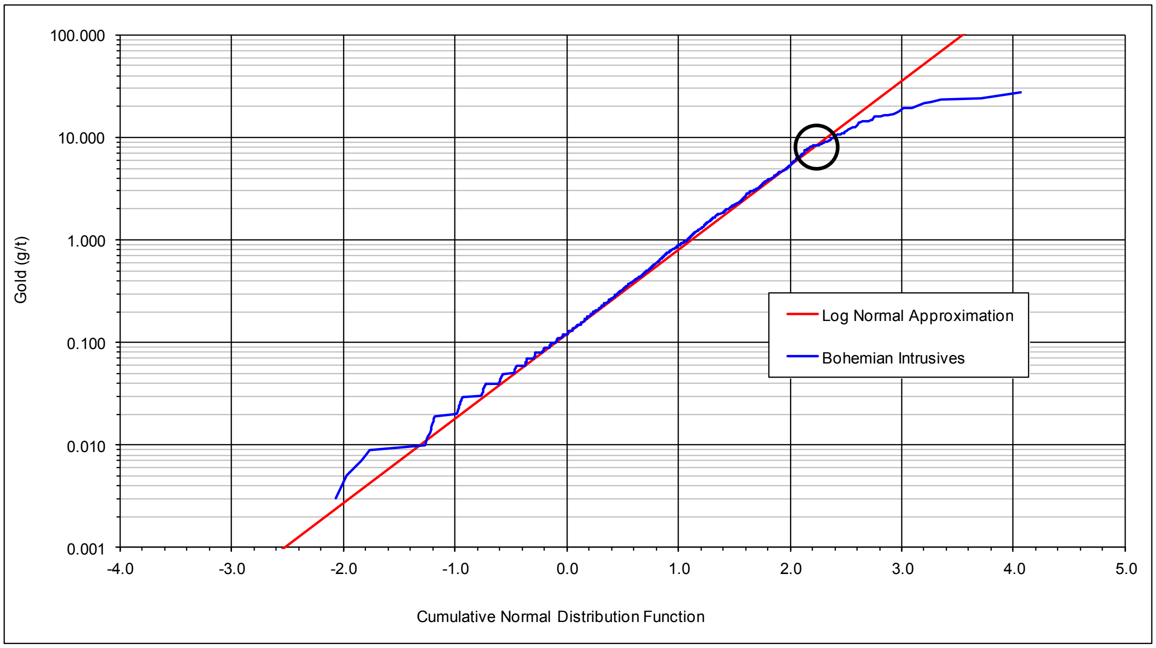
The combined Classic-Lone Star model was rotated 24 degrees (new north axis has an azimuth of 25 degrees) to better accommodate the orientation of the structurally controlled mineralization and to reduce the number of blocks in the model. The combined Classic-Lone Star model contains 558 columns, 100 rows, and 114 levels.

The models were setup to contain a similar number of fields for storing a variety of geologic, topographic, density, and grade data.

14.2.5 Capping of Assays

Isolated high-grade assays, while often substantiated by re-assaying and/or quality assurance-quality control samples, can potentially result in local over estimation of resources. Typically high-grade outlier values are "cut" or "capped" to minimize the potential of over estimating resources. An examination for potential high-grade outlier values was conducted by RMI by analyzing cumulative probability plots and decile/percentile distributions for each deposit by major rock type. Figure 14–16 shows a typical cumulative probability plot for the Bohemian deposit that was used by RMI to identify outliers. The original fire assay results were transformed using the cumulative normal distribution function and then displayed in log normal scale.

Figure 14–16: Au Probability Plot – (Bohemian Intrusives)



Source: RMI (2013)

Similar plots were generated for each mineralized zone for intrusive and sedimentary rocks.

Table 14-18 summarizes the grade capping limits that were used by RMI for each mineralized zone. The raw original assay intervals were capped according to the values shown in Table 14-18 prior to compositing the drillhole data.

Table 14-18: Gold Grade Capping Limits by Area

Area	Au Cap Grade (g/t)	
	Intrusive	Sediment
Bohemian	10.0	5.0
Schooner	10.0	2.5
Lower Fosters	7.5	4.5
West Big Rock	6.0	2.0
East Big Rock	4.0	2.0
Classic	5.0	0.4
Lone Star	5.0	0.3

14.2.6 Compositing

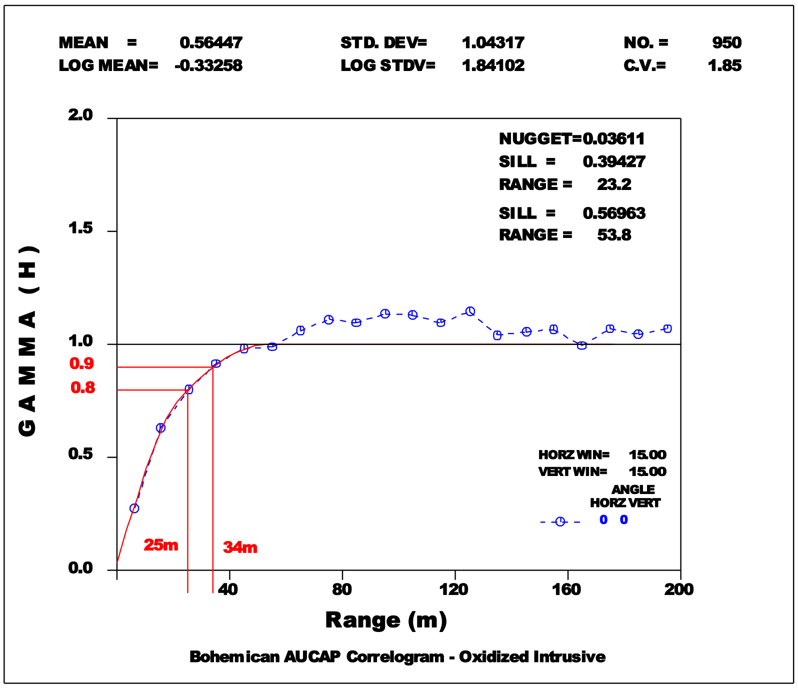
The length of samples from the various drilling campaigns is somewhat variable with many samples in the range of 1.5 to 2.0 metres in length. RMI elected to use 6-metre-long drillhole composites to estimate grades into 6 metre x 6 metre x 6 metre blocks. It is RMI's opinion that the 6-metre-long composites provide appropriate support for estimating grade into 6 metre x 6 metre x 6 metre blocks. The composites contain varying amounts of internal dilution which is appropriate for 6 metre SMU's. Down-hole fixed length composites were generated on six-metre intervals from the collar down the bore hole providing uniform length samples. The compositing routine honored major rock type (intrusive and sedimentary) codes stored in the raw data file, starting and ending the creation of 6-metre-long composites at lithologic contacts.

14.2.7 Variography

RMI generated a variety of grade and indicator variograms for each of the mineralized areas using both MineSight® and Sage2001® software. In general, the grade variograms tended to identify anisotropy in the plane of the mineralized intrusive sills.

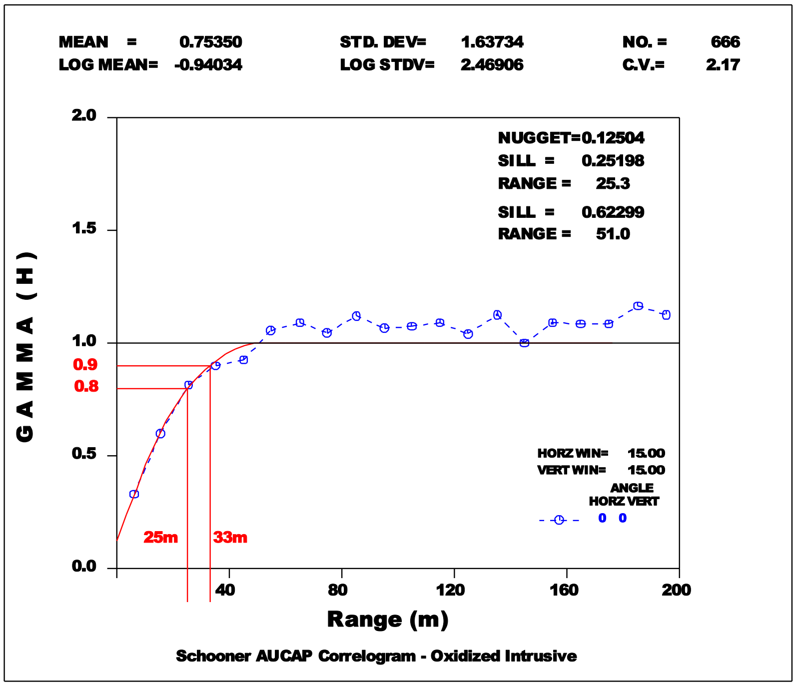
Examples of gold grade correlograms are presented for the Bohemian, Schooner, Lower Fosters, and West Big Rock deposits as Figure 14–17 through Figure 14–20, respectively. These correlograms show nugget effects for these deposits in the range of 0.3 to 0.6. Ranges are indicated at 80% and 90% of the total variance and are shown in red font.

Figure 14–17: Au Grade Correlogram – (Oxidized Bohemian Intrusives)



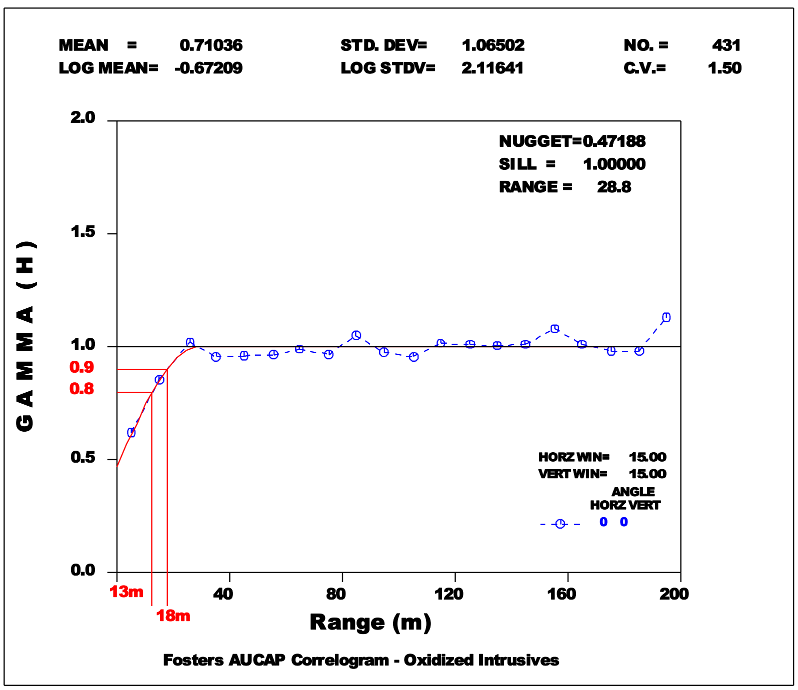
Source: RMI (2013)

Figure 14–18: Au Grade Correlogram – (Oxidized Schooner Intrusives)



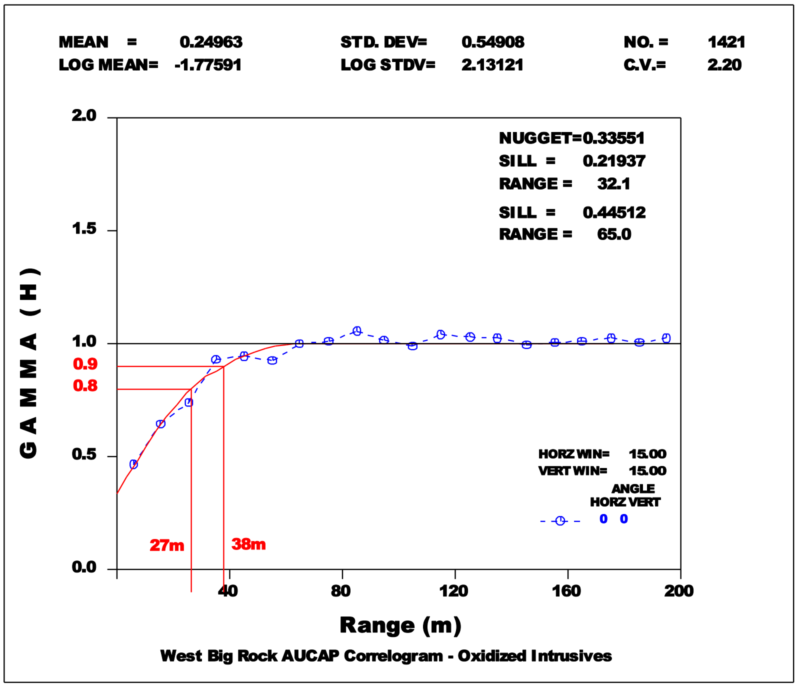
Source: RMI (2013)

Figure 14–19: Au Grade Correlogram – (Oxidized Lower Fosters Intrusives)



Source: RMI (2013)

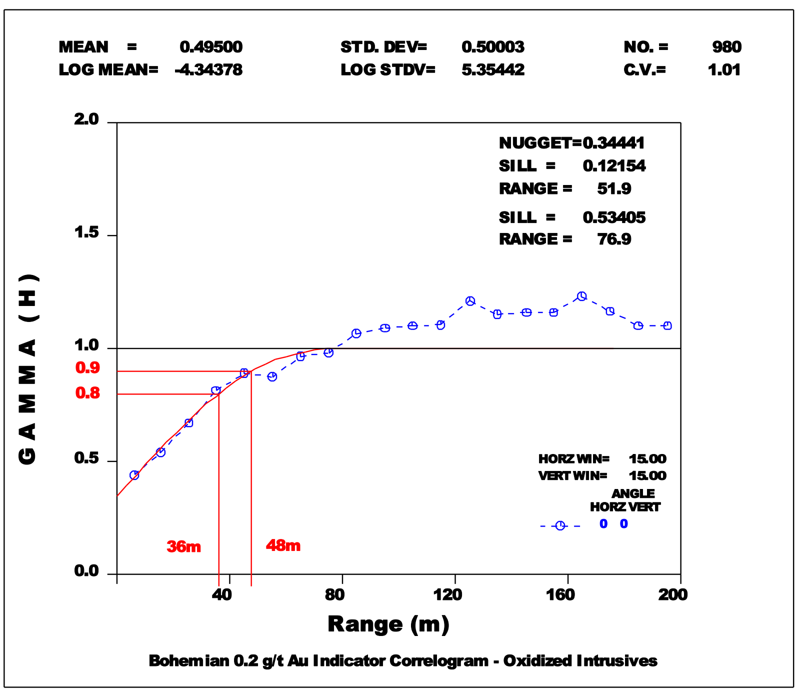
Figure 14–20: Au Grade Correlogram – (Oxidized West Big Rock Intrusives)



Source: RMI (2013)

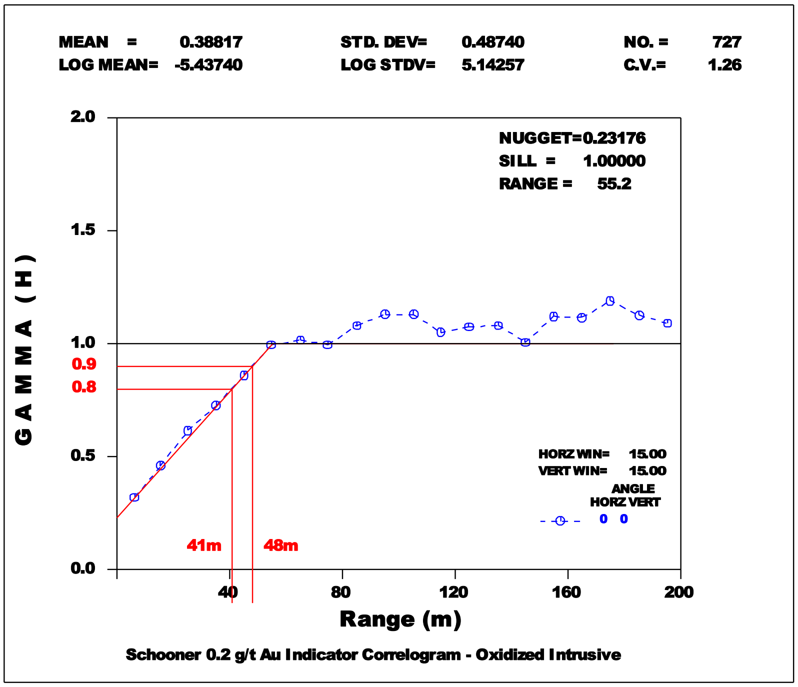
Gold indicator correlograms based on a 0.2 g/t indicator threshold for the Bohemian, Schooner, Lower Fosters, and West Big Rock deposits as Figure 14–21 through Figure 14–24, respectively.

Figure 14–21: 0.2 g/t Au Indicator Correlogram – (Oxidized Bohemian Intrusives)



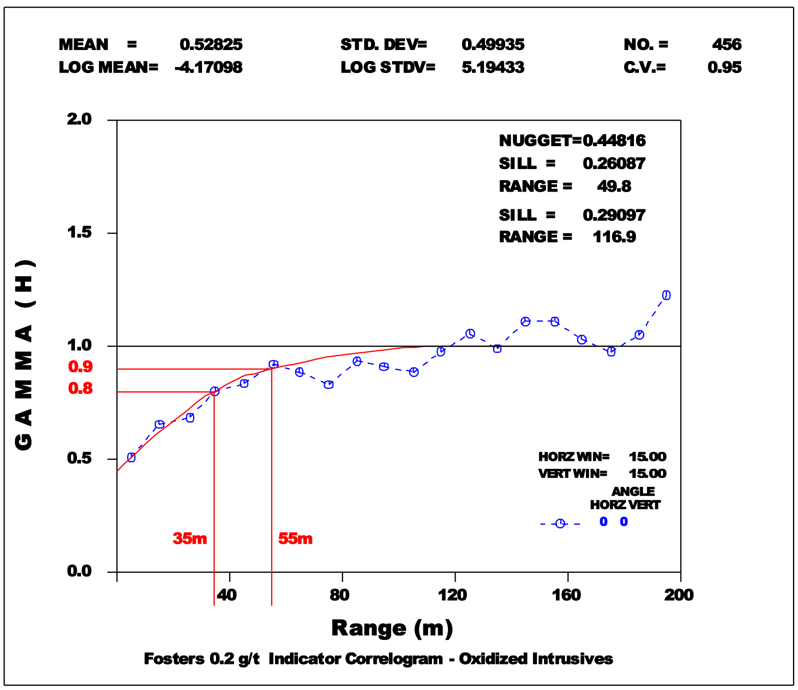
Source: RMI (2013)

Figure 14–22: 0.2 g/t Au Indicator Correlogram – (Oxidized Schooner Intrusives)



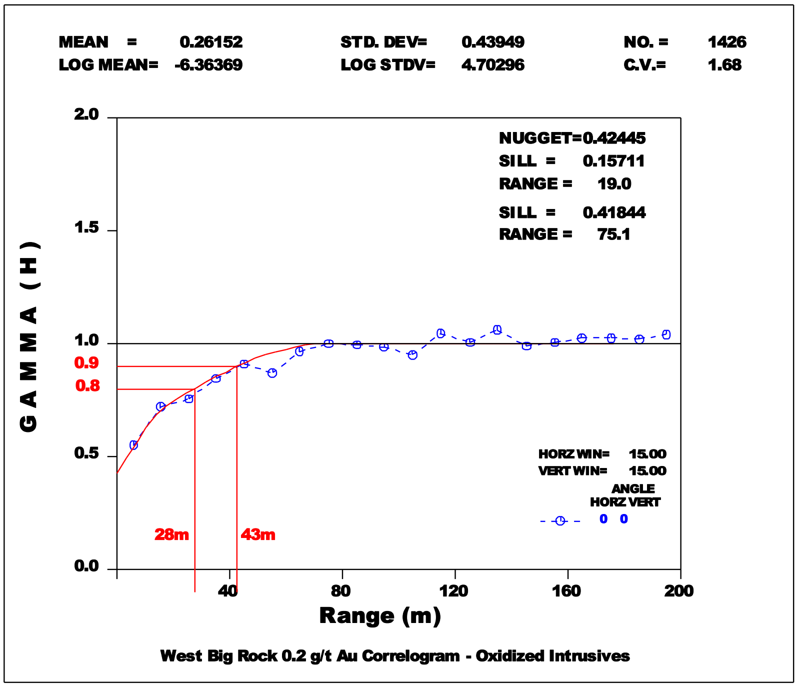
Source: RMI (2013)

Figure 14–23: 0.2 g/t Au Indicator Correlogram – (Oxidized Lower Fosters Intrusives)



Source: RMI (2013)

Figure 14–24: 0.2 g/t Au Indicator Correlogram – (Oxidized West Big Rock Intrusives)



Source: RMI (2013)

14.2.8 Estimation

Block gold grades were estimated for all mineralized areas using an inverse distance cubed method. In general a three or four pass estimation strategy was implemented using a limited number of composites to minimize grade smoothing. Table 14-19 summarizes the main constraints that were used to estimate gold grades for each area.

Table 14-19: Gold Grade Estimation Constraints

Resource Area	Constraint
Bohemian	Intrusive and sediment populations
Schooner	Intrusive and sediment populations
Lower Fosters	Intrusive and sediment populations
West Big Rock	Intrusive and sediment populations by two structural domains
East Big Rock	Intrusive and sediment populations by two structural domains
Classic	Gold grade envelope based on hydrothermal geochem signature
Lone Star	Gold grade envelope based on hydrothermal geochem signature

Table 14-20 lists key parameters that were used for each estimation pass for the various mineralized areas. The Table shows the number of composites used to estimate block grades (i.e. minimum number, maximum, number, and maximum composites per drillhole), the size and orientation of the search ellipse, and whether outlier restriction was used. Outlier restriction does not allow composite grades above a specified value to be projected more than a specified distance.

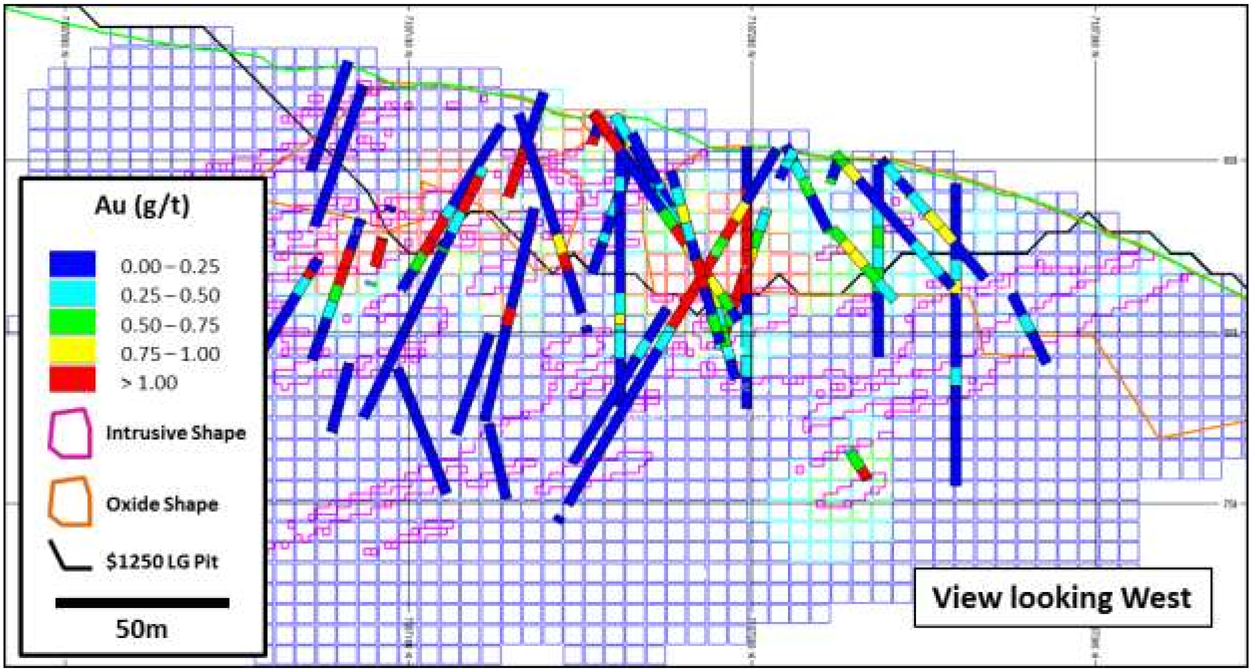
Table 14-20: Gold Grade Estimation Parameters

Resource Area	Pass Number	Number of Composite			Ellipse Range (m)			Ellipse Rotation			Outlier Restriction	
		Min	Max	Max/hole	Major	Minor	Vertical	ROTN	DIPN	DIPE	Au (g/t)	Max Dist (m)
Bohemian	1	1	3	1	4	4	4	75	0	-15	n/a	n/a
	2	3	6	2	37.5	37.5	12.5	75	0	-15	n/a	n/a
	3	3	6	2	75	75	25	75	0	-15	n/a	n/a
	4	1	3	1	25	25	5	75	0	-15	n/a	n/a
Schooner	1	1	3	1	4	4	4	90	0	-15	n/a	n/a
	2	1	3	1	25	25	5	90	0	-15	n/a	n/a
	3	1	3	1	50	50	10	90	0	-15	n/a	n/a
Lower Fosters	1	1	3	1	4	4	3	90	0	-35	n/a	n/a
	2	1	3	2	25	25	12.5	90	0	-35	n/a	n/a
	3	1	3	2	50	50	25	90	0	-35	n/a	n/a
West Big Rock	1	1	3	1	4	4	3	70	0	-35	3	12
	2	2	3	1	25	25	5	70	0	-35	3	12
	3	2	3	1	50	50	10	70	0	-35	1.5	12
	4	1	3	1	25	25	5	70	0	-35	1.5	12
East Big Rock	1	1	3	1	4	4	3	120	0	0	3	12
	2	2	3	1	25	25	5	120	0	0	3	12
	3	2	3	1	50	50	10	120	0	0	1.5	12
	4	1	3	1	25	25	5	120	0	0	1.5	12
Classic	1	1	3	1	4	4	3	100	0	-55	n/a	n/a
	2	1	3	1	37.5	37.5	5	100	0	-55	n/a	n/a
	3	1	3	1	75	75	10	100	0	-55	n/a	n/a
	4	1	3	1	100	100	15	100	0	-55	n/a	n/a
Lone Star	1	1	3	1	4	4	3	100	0	-55	2	12
	2	1	3	2	37.5	37.5	5	100	0	-55	2	12
	3	1	3	2	75	75	10	100	0	-55	2	12

14.2.8.1 Model Validation

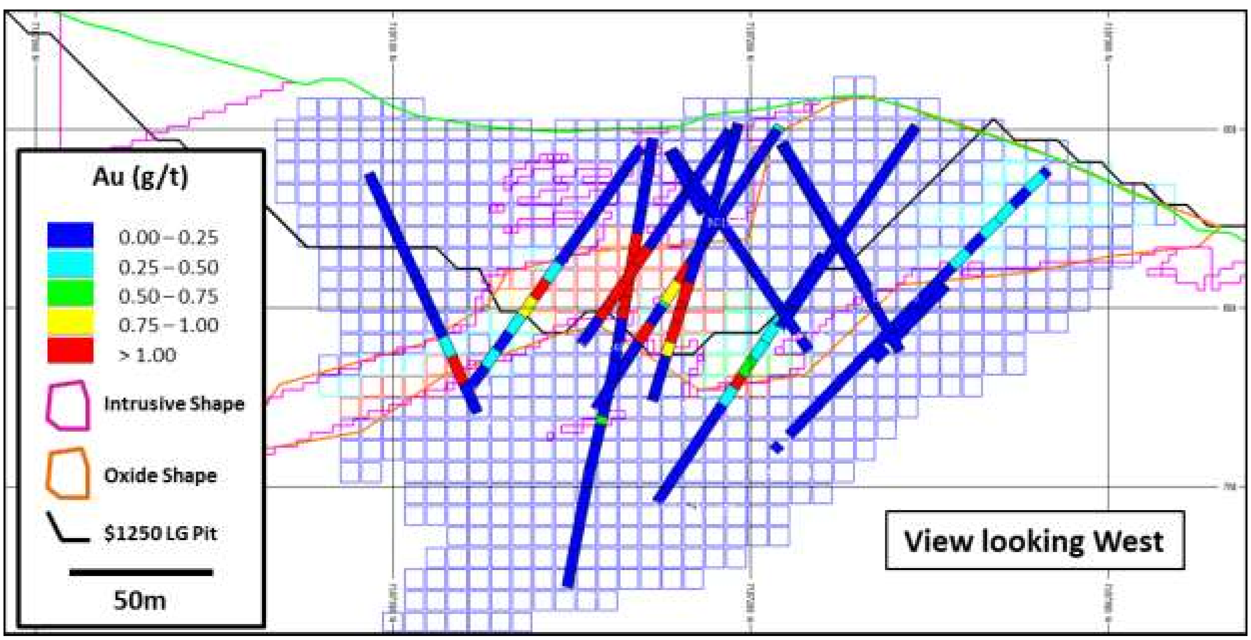
The grade models were validated by visual and statistical methods. The estimated block grades were compared against drillhole composites in both sectional and level plan views. In the opinion of RMI, there is a close comparison between block and drillhole composite grades. Figure 14–25 through Figure 14–34 are representative cross sections and cross section locations that compare drillhole composites with model blocks for the Bohemian, Schooner, Lower Fosters, West Big Rock, East Big Rock, and Classic deposits, respectively. Conceptual pit outlines are shown on each cross section as heavy black lines.

Figure 14–25: Bohemian Block Model Section A-A' (See Figure 14 27 for Location)



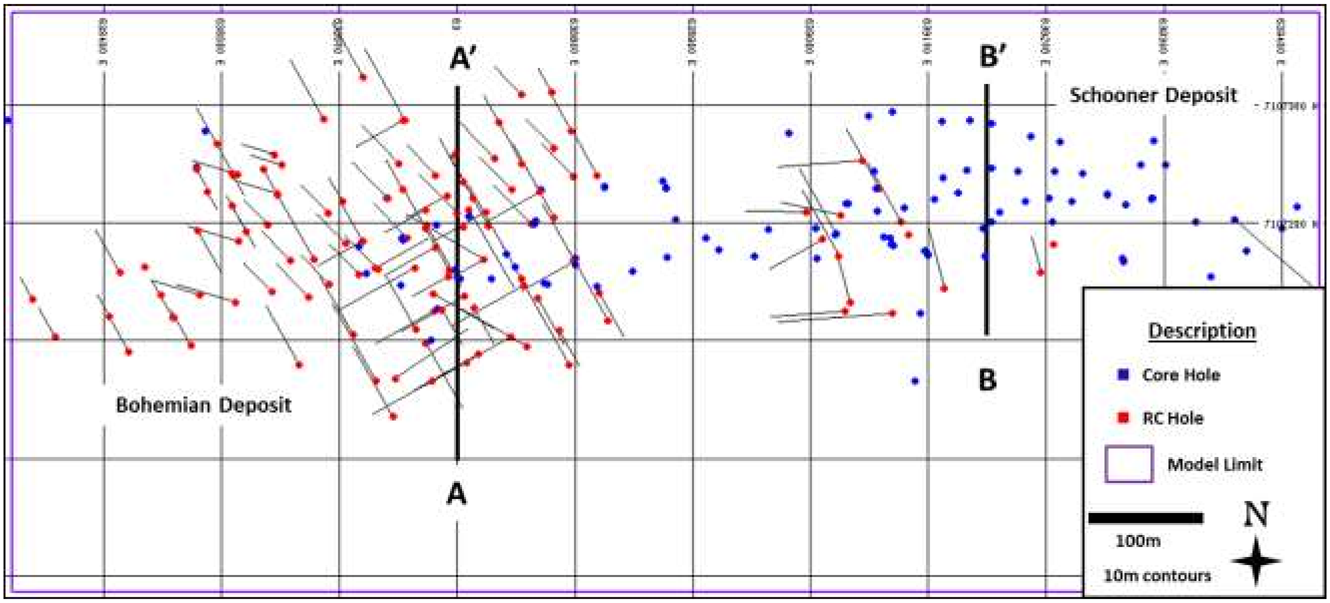
Source: RMI (2013)

Figure 14–26: Schooner Block Model Section B-B' (See Figure 14 27 for Location)



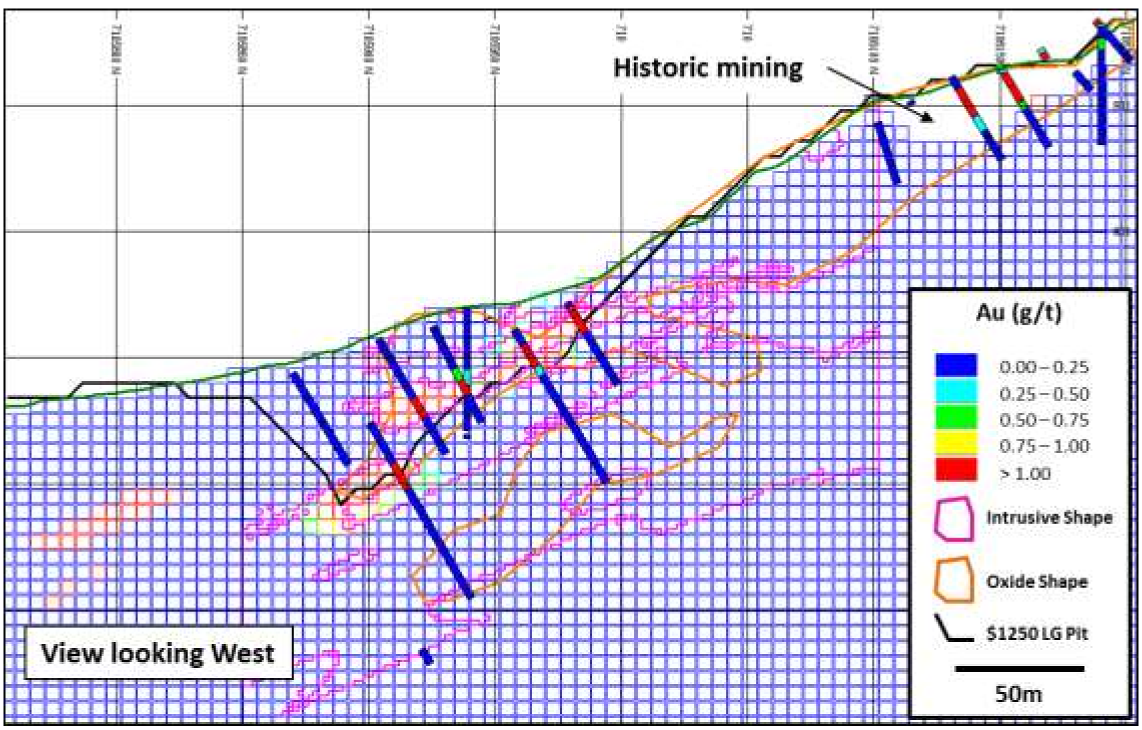
Source: RMI (2013)

Figure 14–27: Location of Bohemian (A-A') and Schooner (B-B') Cross Sections



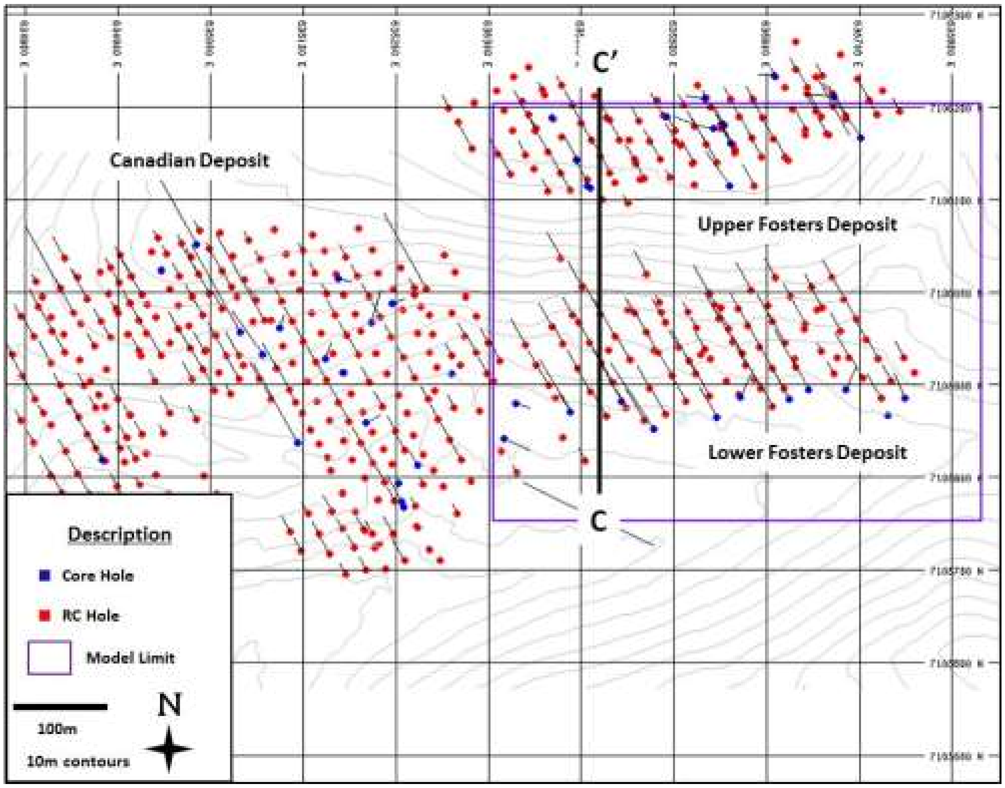
Source: RMI (2013)

Figure 14–28: Lower Fosters Block Model Section C-C' (See Figure 14 29 for Location)



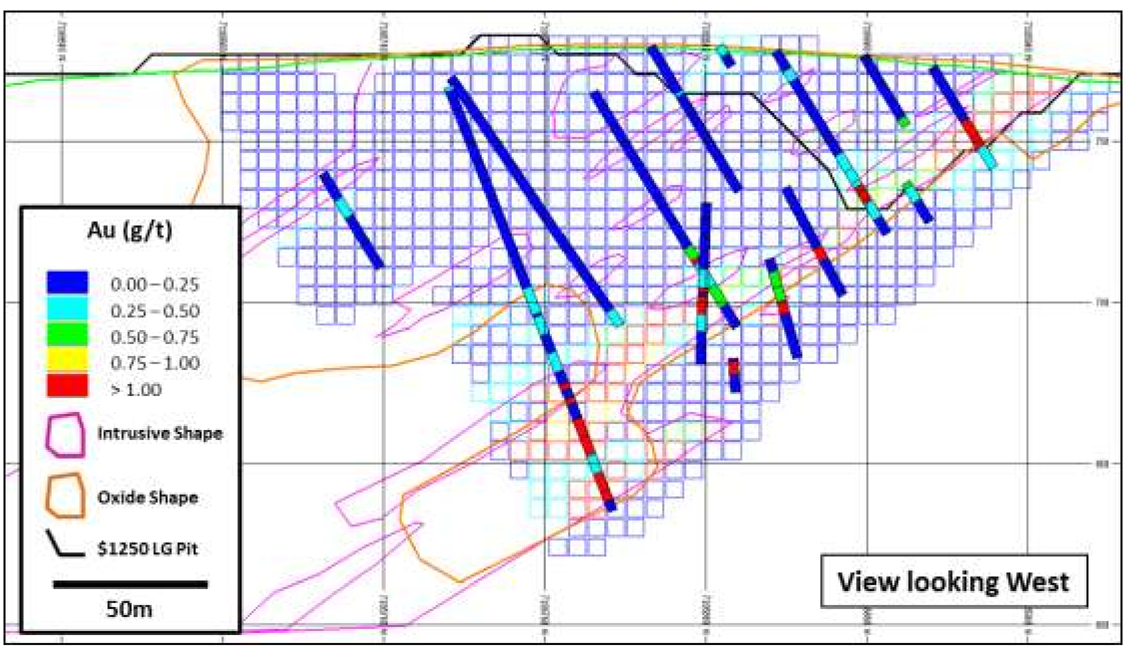
Source: RMI (2013)

Figure 14–29: Location of Lower Fosters (C-C') Cross Section



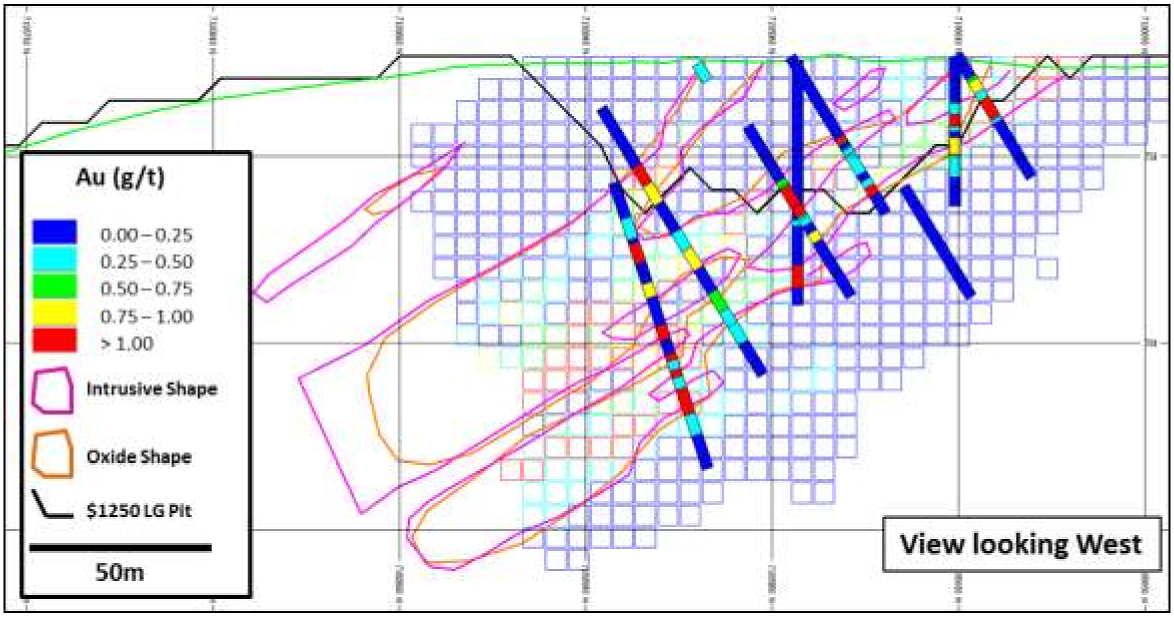
Source: RMI (2013)

Figure 14–30: West Big Rock Block Model Section D-D' (See Figure 14 32 for Location)



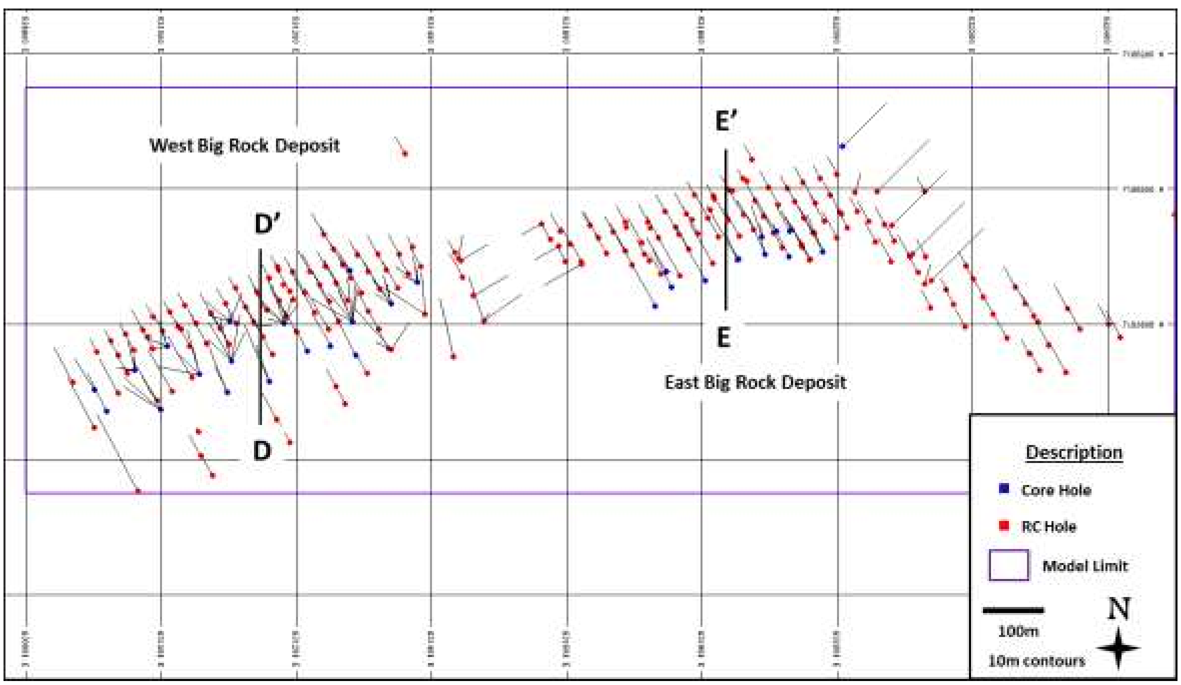
Source: RMI (2013)

Figure 14–31: East Big Rock Block Model Section E-E' (See Figure 14 32 for Location)



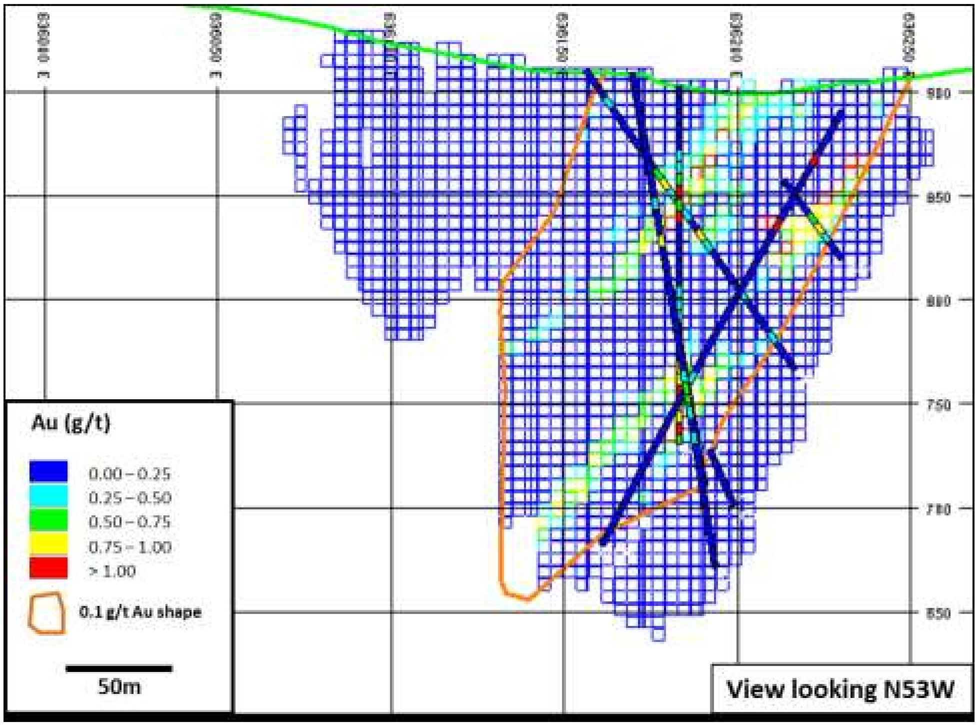
Source: RMI (2013)

Figure 14–32: Location of West Big Rock (D-D') and East Big Rock (E-E') Cross Sections



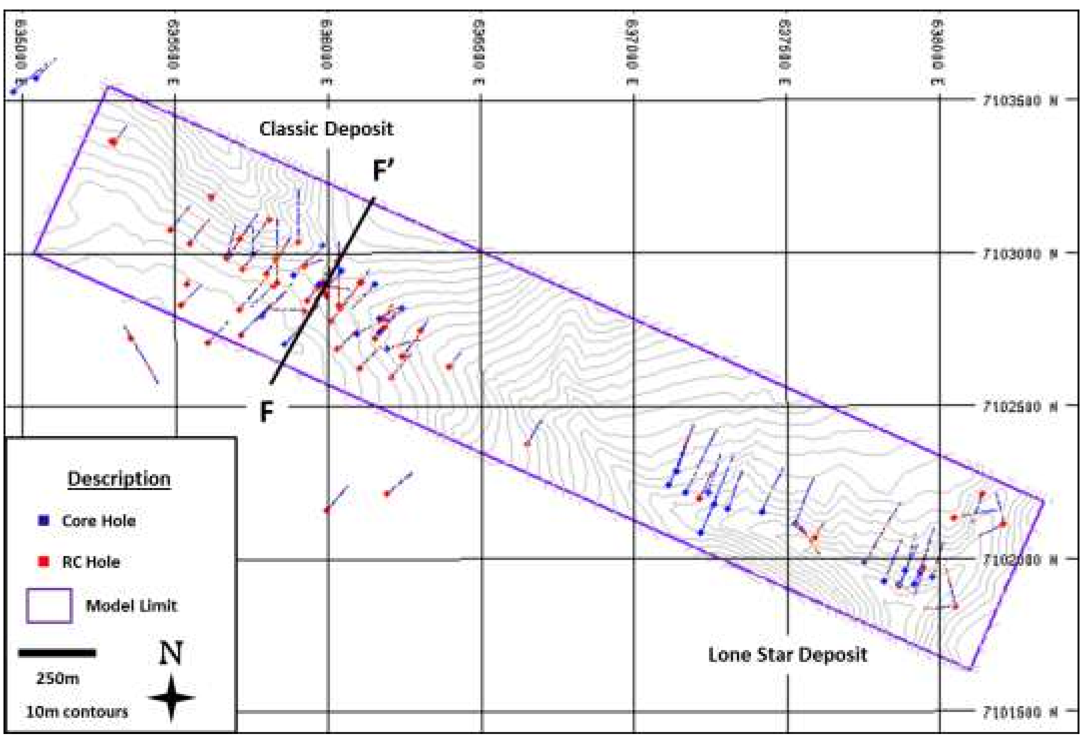
Source: RMI (2013)

Figure 14–33: Classic Block Model Section F-F' (See Figure 14 34 for Location)



Source: RMI (2013)

Figure 14–34: Location of Classic (F-F') Cross Section



Source: RMI (2013)

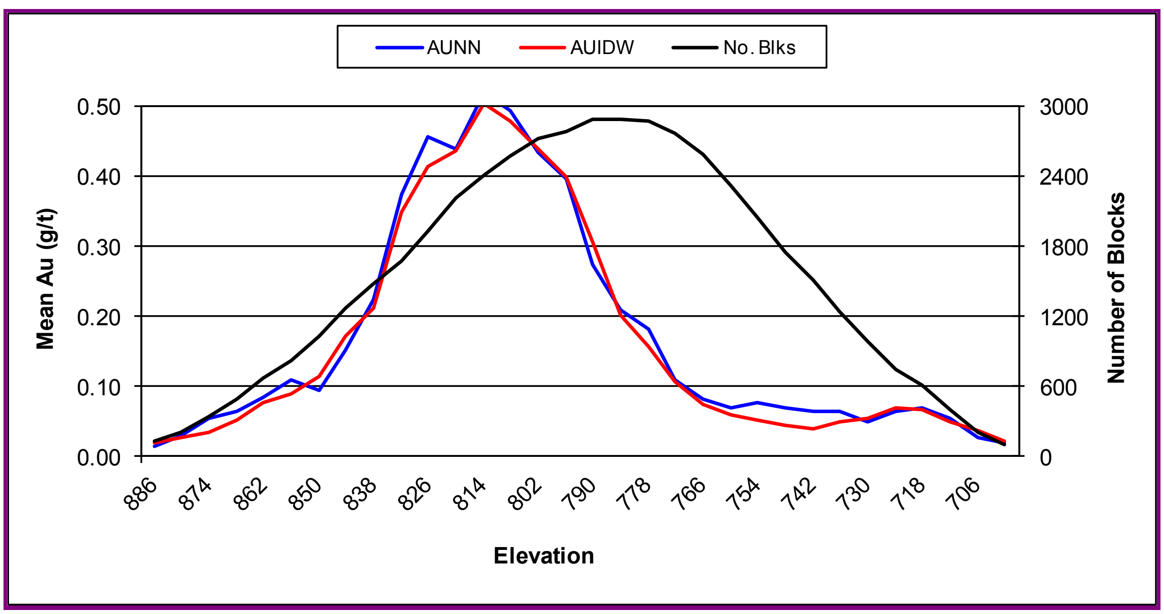
Nearest neighbor gold grade models were constructed to check for potential global biases in the inverse distance grade models. Table 14-21 compares the inverse distance (IDW) grade with a nearest neighbor (NN) grade using a zero cut-off grade. The comparisons are shown for both Indicated and Inferred resources. Several of the deposits show a slight low bias with regards to the inverse distance grade. RMI believes that this is not material given the intercalated nature of the mineralized intrusive sills and often unmineralized sedimentary rocks.

Table 14-21: Global Bias Check – Inverse Distance vs. Nearest Neighbor Grades

Resource Area	Indicated Resource			Inferred Resource		
	IDW	NN	% Diff	IDW	NN	% Diff
Bohemian	0.224	0.2317	-3.30%	0.0809	0.0812	-0.40%
Schooner	0.2612	0.2576	1.40%	0.156	0.1589	-1.80%
Lower Fosters	0.2172	0.2287	-5.00%	0.0991	0.1019	-2.70%
West Big Rock	0.1566	0.1663	-5.80%	0.09	0.0897	0.30%
East Big Rock	0.1345	0.1446	-7.00%	0.0769	0.0717	7.30%
Classic	n/a	n/a	n/a	0.1496	0.1463	2.00%
Lone Star	n/a	n/a	n/a	0.1187	0.1188	-0.20%

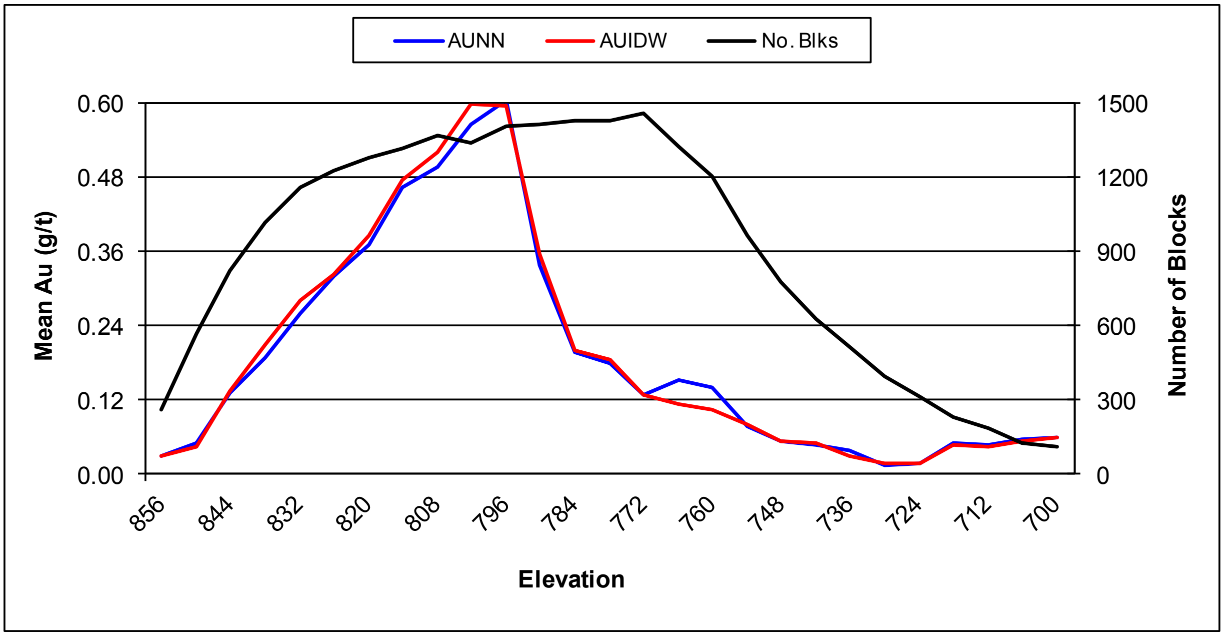
Local bias checks were made by generating a series of "swath" plots through the block model. These plots compare the inverse distance and nearest neighbor grade models as vertical slices (east-west and north-south) and horizontal slices (level plans) through the block model. Figure 14–35 through Figure 14–41 show level plan slices through the Bohemian, Schooner, Lower Fosters, West Big Rock, East Big Rock, Classic, and Lone Star models, respectively. Note that only Indicated blocks are depicted for Figure 14–35 through Figure 14–39, while only Inferred blocks are summarized in Figure 14–40 and Figure 14–41.

Figure 14–35: Bohemian Gold Swath Plot by Elevation Levels



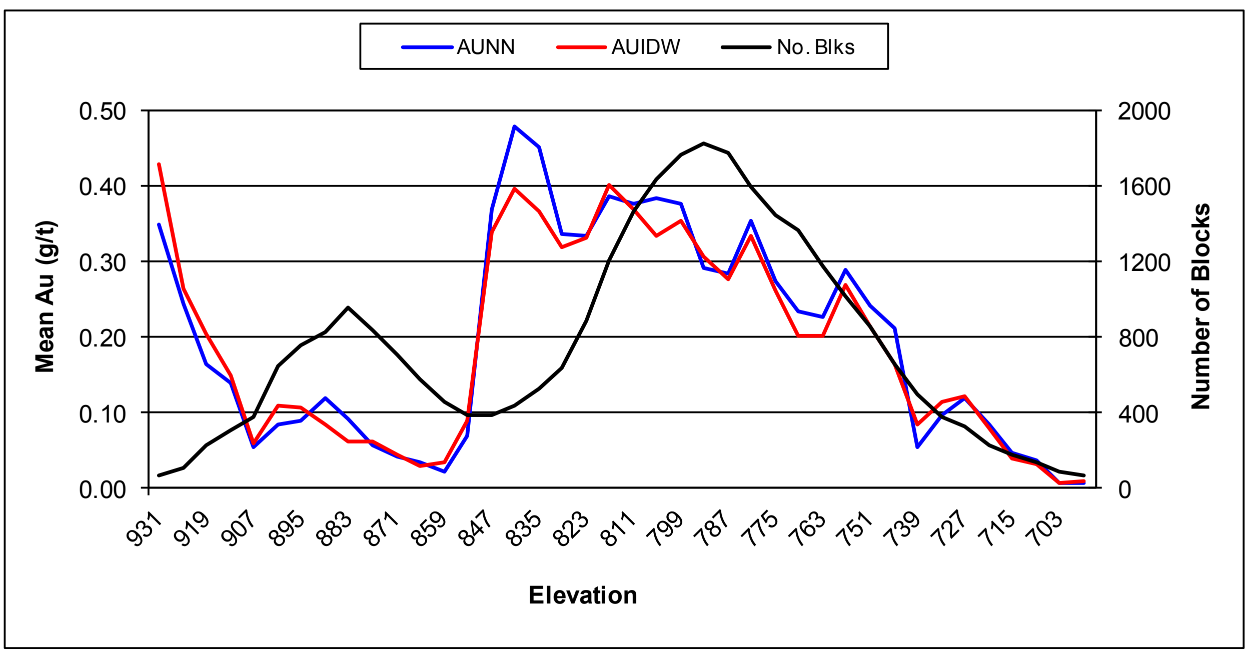
Source: RMI (2013)

Figure 14–36: Schooner Gold Swath Plot by Elevation Levels



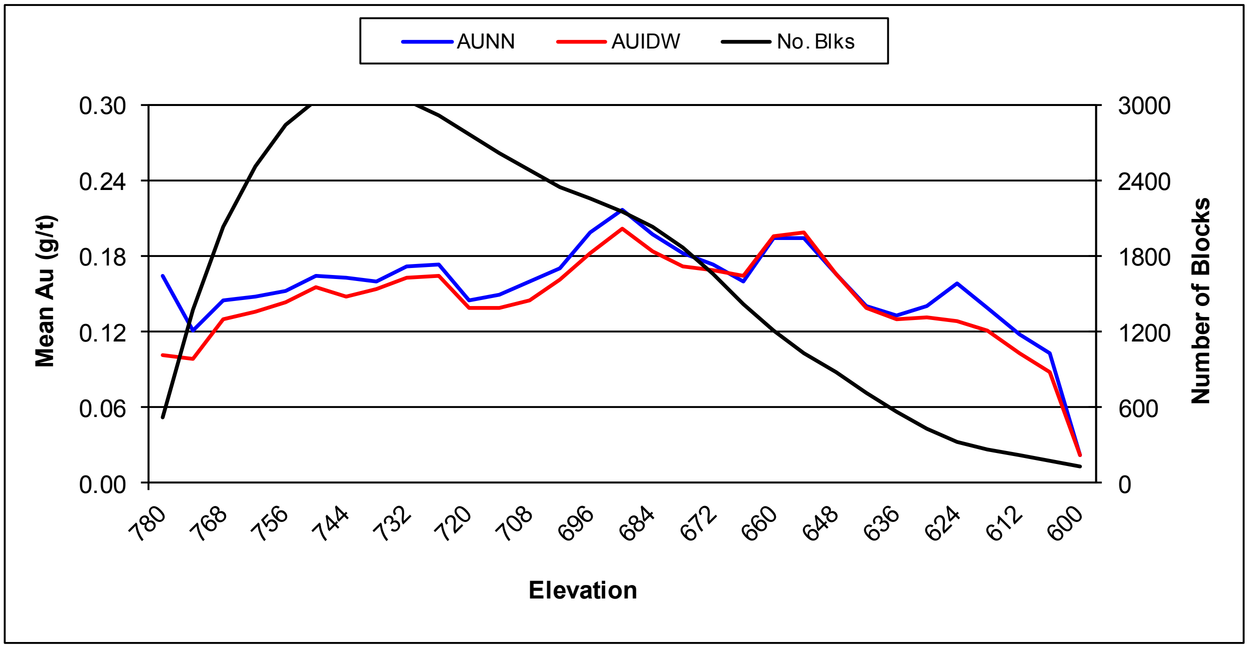
Source: RMI (2013)

Figure 14–37: Lower Fosters Gold Swath Plot by Elevation Levels



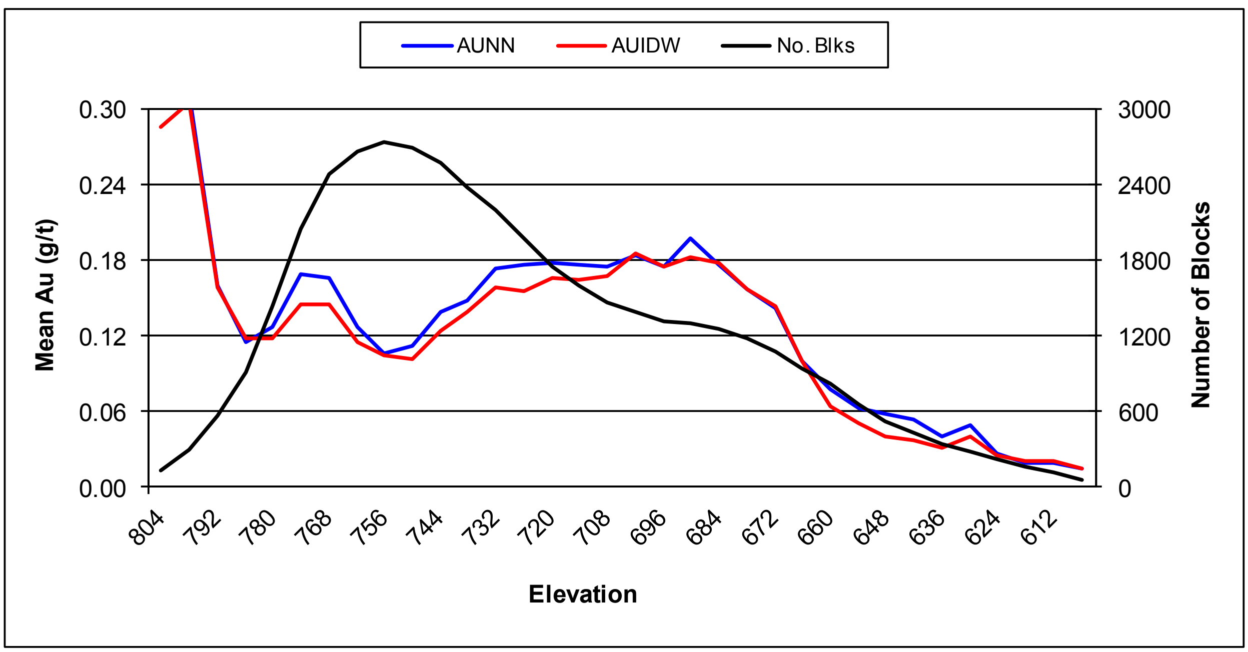
Source: RMI (2013)

Figure 14–38: West Big Rock Gold Swath Plot by Elevation Levels



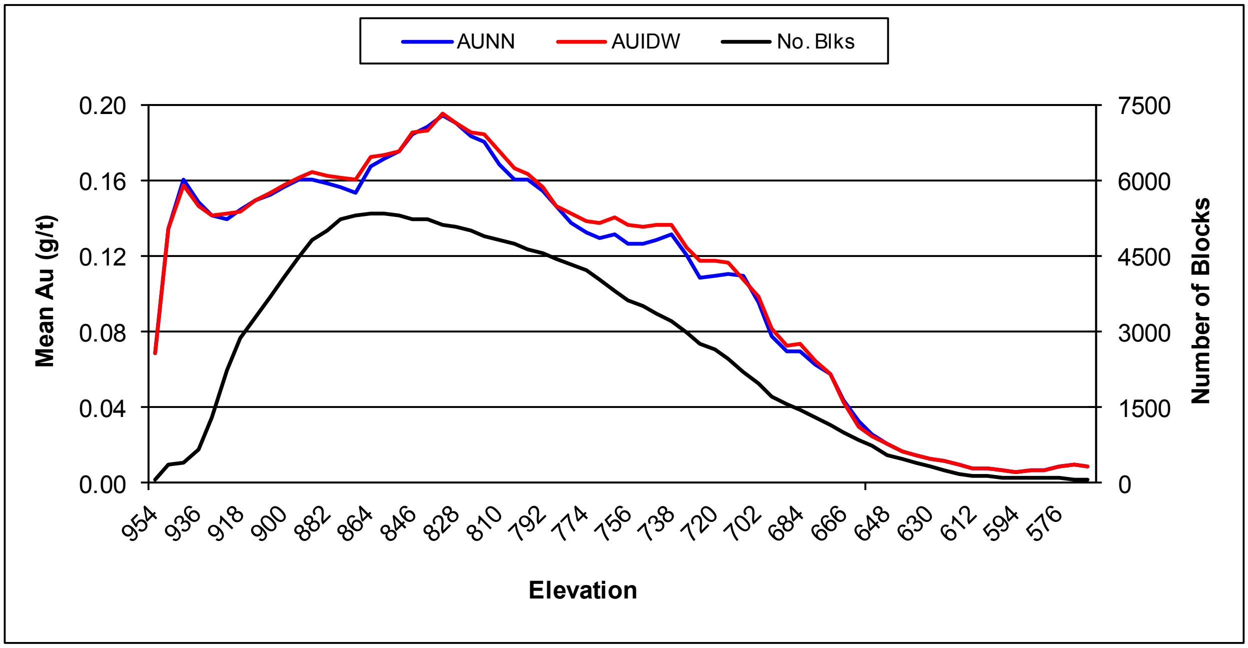
Source: RMI (2013)

Figure 14–39: East Big Rock Gold Swath Plot by Elevation Levels



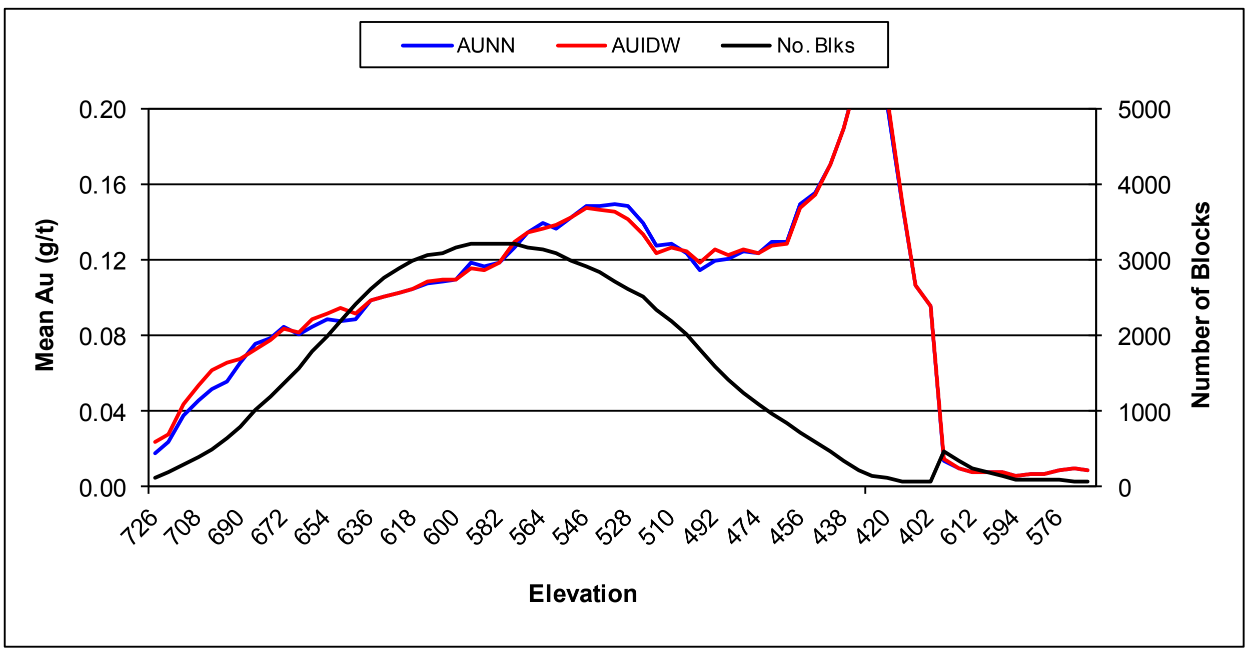
Source: RMI (2013)

Figure 14–40: Classic Gold Swath Plot by Elevation Levels



Source: RMI (2013)

Figure 14–41: Lone Star Gold Swath Plot by Elevation Levels



Source: RMI (2013)

14.3 Historical Heap Leach Pad

Viceroy Minerals operated an open pit and run-of-mine heap leach operation at the Brewery Creek Project from approximately 1996 through September 2002. Monthly report data indicate that Viceroy placed about 10.4 million tonnes on the heap leach pad at an average gold grade of about 1.5 g/t containing approximately 502,000 ounces of gold in situ. Based on Viceroy reports about 279,000 ounces were produced from the heap leach pad.

Based on that information, Golden Predator undertook a sonic drilling program to collect samples from the Viceroy leach pad. A total of 18 four inch diameter sonic holes were drilled in 2011 on approximately 100 m centers. The holes were sampled on five-foot (1.52m) intervals resulting in 177 samples which were analyzed at McClelland Laboratories located in Reno, Nevada. The average head grade, established by conventional fire assay methods, was approximately 0.66 g/t which correlates well with the calculated residual contained grade of 0.59 g/t. McClelland undertook additional test work including cyanide soluble analyses along with preg robbing characteristics.

The sonic drillhole samples were combined at the McClelland Lab to create material for four column leach tests. A total of 28 composites were generated from the sonic samples. The composites were crushed to 80% passing 9.5mm and then subjected to 96 hour bottle roll tests. SGS Metcon from Tucson, Arizona estimated gold recovery from the four column tests to be about 47.5% after 141 days of leaching.

RMI constructed a 3D block model of the heap leach pad and estimated grades using the sonic drillholes. A basal surface was generated approximately 3 metres above the synthetic liner. No block grades were estimated above the old heap leach liner and the below the protective buffer zone surface. There is approximately 905,000 m³ (about 1.5 million tonnes) of material above the liner and below the described protective boundary surface.

A block size of 3m x 3m x 3m was selected along with 3-metre-long composites. Fire assay and cyanide soluble gold grades were estimated using a three pass inverse distance method. A high inverse distance power of five was used based on comparisons with a nearest neighbor model. The first pass used a large search ellipse (300 m x 300m x 50m) to ensure that all blocks were estimated. The second pass used a search strategy of 125 m x 125m x 21 m. The last pass used a search ellipse of 75m x 75m x 12m. Previously estimated blocks were overwritten by subsequent tighter search ellipse runs. A maximum of three samples were allowed to estimate the blocks. This strategy resulted in a more "polygonal" estimate but, in the opinion of RMI, this is appropriate for this project.

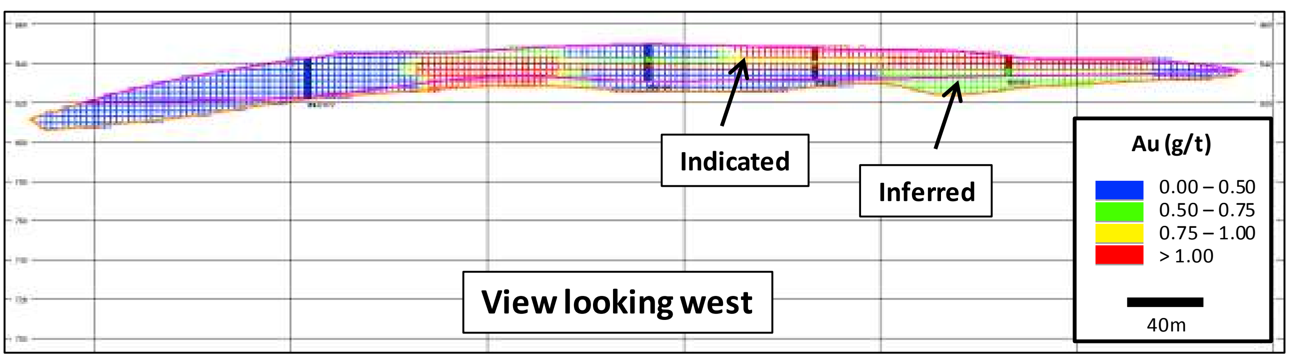
A bulk density value of 1.70 g/cm³ was used to tabulate tonnages. This density was derived by test work that was completed by Viceroy and seems to be reasonable for run-of-mine truck dumped primarily intrusive material.

Figure 14–42 is a plan view showing the existing heap leach pad, the 18 sonic hole locations (red dots) and a line of section (A-A') for Figure 14–43, which is a north-south trending vertical cross section. A surface was constructed 3 metres above the heap leach liner and no grades were estimated below the buffer surface. Another surface was constructed at the base of the sonic holes which were intentionally drilled short of the liner as a precaution against compromising the liner.

Figure 14–42: Plan View of Viceroy Heap Leach Pad Showing Sonic Holes

Source: Golden Predator (2013)

Figure 14–43: Heap Leach Cross Section A-A'



Source: RMI (2013)

14.4 North Slope and Sleeman Deposits

Mr. James F. Barr, P.Geo., Senior Geologist with Tetra Tech EBA reported mineral resource estimates for the North Slope (NS) and Sleeman (SL) deposits as part of the 2013 Technical Report titled "Updated mineral Resources Estimate for the Brewery Creek Property" prepared for Golden Predator Corp., effective March 11, 2012, and amended on January 17, 2013. The resource estimates for both the North Slope and Sleeman deposits remain current, however, were not included as part of the PEA as their mineral resource estimates did not meet the current objectives of Golden Predator.

The estimates for NS and SL were originally reported using a 0.2 g/t Au as the base case cut-off for oxide resources in the previous Technical Report. The base case is being revised to use the 0.5 g/t Au cut-off reported in the Technical Report's sensitivity tables to better reflect current market conditions.

The following sections summarize the information contained in the January 17, 2013 Technical Report.

14.4.1 Deposit Geology Pertinent to Resource Estimation

Geological solids were constructed to represent the major lithologies identified at each of the deposit areas and used in the creation of a geological model within the block model using Geovia Gems® v6.4.1 (Gems). In most respects, the lithologies for each deposit were simplified due to the complexity of individual rock codes used in the drillhole database. The geological model was used to define rock codes (Table 14-22) to each individual block using a partial block percentage methodology.

Table 14-22: Descriptions for Major Rock Types used in Geologic Model

Type	Major Rock Type	Major Rock Code	Description
Oxidized	LAQM	121	Limonite altered quartz monzonite
	SED_OX	220	Argillite, graphitic argillite, siltstone with limonite staining
Unoxidized	AQM	143	Altered quartz monzonite
	SED_NX	221	Argillite, graphitic argillite, siltstone
Other	OB	601	Overburden
	Waste	888	Unknown, unmineralized material
	Air	999	Air

The North Slope deposit includes a single semi-continuous zone of mineralization hosted primarily within sedimentary rocks with higher grade mineralization occurring in proximity to a few thin quartz-monzonite intrusive sills. Mineralization within the sedimentary rocks occurs along similar orientation to these sills. The sedimentary and intrusive rock units respected the interpreted redox boundary that was incorporated into the geological model, as described below. The model incorporated numerous geological and mineralization domains used for modeling of gold grade values into the block model. In general, the mineralization was found to be continuous in distinct shear packages within the sedimentary host lithologies.

The Sleeman Deposit includes a single semi-continuous zone of mineralization distributed along sub-vertical fault bound pathways and lower angle stratiform quartz-monzonite sills. The Sleeman deposit was modeled using both sedimentary and intrusive quartz-monzonite rock types as the two primary lithologies; however, gold mineralization was constrained to the quartz-monzonite unit. The quartz-monzonite intrusive was subdivided into LAQM and AQM respecting the redox boundary interpreted for the area, as described below. Sedimentary 'selvages' modeled to occur within the quartz-monzonite sill were noted to be barren of mineralization.

Both geological models were bound at the surface by LIDAR data provided by Golden Predator. Natural overburden and till was considered insignificant at these deposits and was ignored in the block model.

14.4.1.1 Oxidation

The redox boundary was provided as a geological surface and was incorporated into the geological model based upon the assumption that gold-bearing mineralization occurring above the boundary is oxidized and material below the surface is unoxidized or hosted within a sulfide phase. The boundary was interpreted based on either visual geological coding from recent Golden Predator drilling or from rock identification within historical drill logs where no recent Golden Predator drilling exists. The redox scheme that was used in the field by company geologists applied an incremental scale for visual observation from 0 to 4, where 0 described unoxidized material and 4 described completely oxidized material. The interpreted geological contact lying between rocks identified as 2 and 3, describing weak and partial oxidation respectively, was typically chosen as the redox boundary. No transition zone was defined at this time. All material coded as sedimentary was exempt from this redox distinction and was modeled as “unoxidized” for the purposes of resource reporting as Tetra Tech EBA felt historical data from Viceroy operations suggested the materials have the potential for preg-robbing and as such may not react similarly to the oxidized quartz-monzonite material. As a result, sedimentary rocks were subject to higher grade cut-offs for resource reporting. Sedimentary rocks at North Slope are an exception to this and were modeled to respect the interpreted oxide boundary by reporting sedimentary resources as oxidized and unoxidized.

14.4.2 Data Used for Estimation

Drillhole data used in the resource was provided by Golden Predator in a database format which included details on header, survey, analytical, lithological, mineralogical, and alteration. The complete drillhole database includes 2,432 holes, of which 90 core holes and 115 RC holes were geographically subset for use in the modeling based on the proximity to the target areas of interest (Table 14-23). The database subset was reviewed by Tetra Tech EBA and corrections were made in collaboration with Golden Predator, where necessary.

Table 14-23: Drillholes by deposit used in the Mineral Resource Estimate

Deposit	Number of Drillholes	Total Metres Drilled
North Slope	140	24,323.11
Core	32	6,657.14
RC	108	17,665.97
Sleeman	65	11,373.83
Core	58	10,871.83
RC	7	502
Total	205	35,696.94

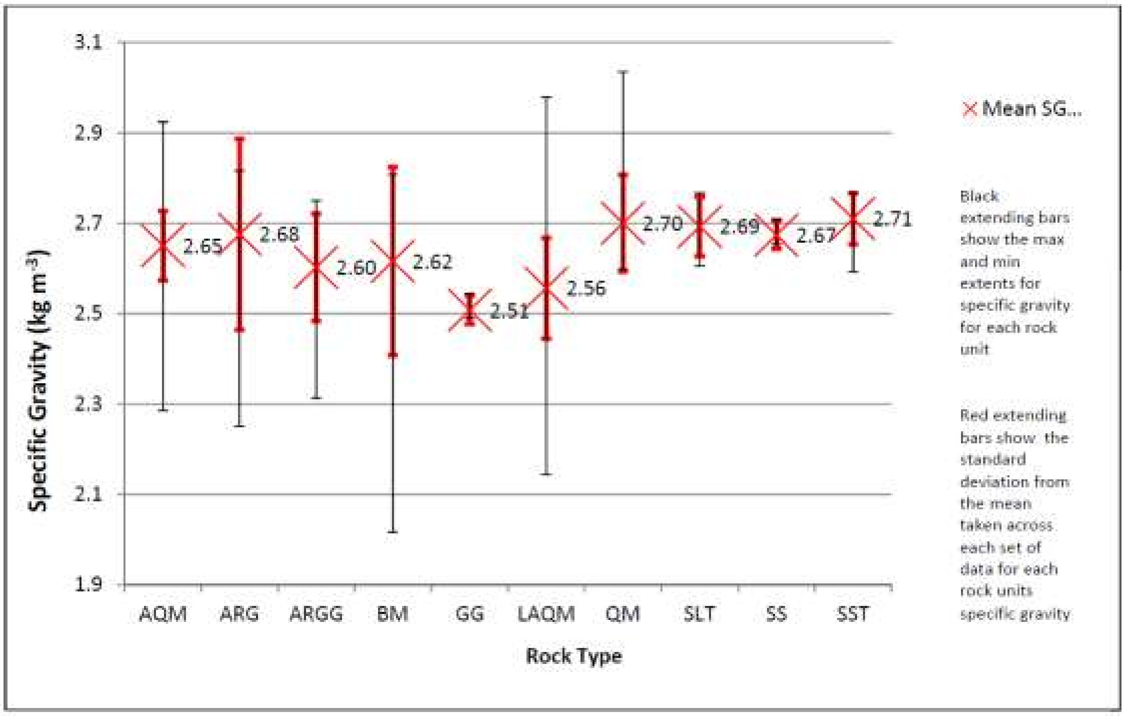
14.4.3 Bulk Density

In situ testing performed by Golden Predator on drill core samples during the 2011 drilling campaign resulted in 467 specific gravity determination (SG) values obtained by in-situ wet density methods. The individual results were correlated to lithology by Tetra Tech EBA and then inspected to obtain a representative value for each lithology. The analysis was conducted by calculating the mean specific gravity values for the available data. The extents of SG and variability in SG high hand low values were plotted against the rock units the SG data represented.

Some manipulation of the dataset was applied and is described as follows. Data values lower than a value of 1 were considered to be anomalous and were omitted from the analysis to avoid a bias in the skewness of the mean. Data that had SG values greater than 5 without mineralogical support were also considered anomalous and were omitted from the analysis.

The results of the analysis are plotted in Figure 14–44 and summarized in Table 14-24 for the major lithologies used in the geological model. Results of specific gravity tests on verification samples collected by Tetra Tech EBA in March 2012, discussed in Section 12, conform well to the values determined from the average in situ Golden Predator test work.

Figure 14–44: Specific Gravity Determination by Rock Type



Source: EBA (2013)

Table 14-24: Specific Gravity Values used for North Slope and Sleeman

Rock Type	Specific Gravity used in Geological Model
LAQM	2.56
AQM	2.65
SED	2.67
WASTE	2.50
OVB	2.50
OTHER	2.50

14.4.4 Methodology

A summary of block modeling parameters used for the North Slope and Sleeman Mineral Resource estimates is included below.

The Mineral Resource estimates were performed using geological and block models in Geovia Gems® v6.4.1. Block size selected for the models were 6x6x6 metres. Block model origins were selected to include sufficient waste, or unmineralized material, around the spatial limits of the interpreted mineralized zones, both vertically and horizontally. Table 14-25 shows the block model parameters.

Table 14-25: Block Model Origins and Dimensions

Resource Area		Origin (UTM m)	Number of Blocks	Block Size (m)	Rotation
NS	X	6634798	170	6	0
	Y	7106700	135	6	0
	Z	552	105	6	0
SL	X	640900	140	6	0
	Y	7105542	90	6	0
	Z	540	65	6	0

For the North Slope model, the majority of the gold grade was constrained to a 0.1 g/t Au grade shell in addition to some loosely constrained mineralized zones outside of the grade shell within the wall rock. All blocks containing gold grade were controlled using ellipse ranges not exceeding the ranges supported by variography for their respective domain.

Modeling of gold grade values into the block model at Sleeman was constrained by two 0.5 g/t Au grade shells within a sub-vertical fault and within loosely constrained geological solids within the stratiform quartz-monzonite sills. All blocks containing gold grade were controlled using ellipse ranges not exceeding the ranges supported by variography for their respective domain. Continuity is supported by visual interpretation of the geological and grade solids, ellipse orientation and range, and with filtering criteria used for the Classification of Mineral Resources. Pit constraints were not applied to either the North Slope or Sleeman model.

14.4.5 Capping of Assays

Initial analysis of the gold grade log-histogram distributions for each deposit area indicate that grade populations are positively skewed and are generally contain few high grade outliers. Using Gems, Tetra Tech EBA visually scrutinized the grade distributions using composited drillhole data and determined that many high grade composites lay with areas of high mineral concentrations. A handful of composite samples were considered to be truly anomalous and were subjected to a high grade cap within the grade interpolation process. Table 14-26 lists the grade caps applied to the composited database to each deposit before interpolation and the actual number of samples that were subjected to the capping.

Table 14-26: High Grade Caps Applied to Composites

Resource Area	Composite Capping Grade (Au g/t)	Number of Samples Capped
North Slope	15	0
Sleeman	16	1

14.4.6 Compositing

A composite length of 2 metres was selected based on the population median of the sample length histogram analysis to normalize the data before being subject to geostatistical analysis. A summary of the raw and composited data with the related descriptive statistics is presented in Table 14-27. Minimal smoothing of the raw data resolution was noted in the 2 metres composite dataset.

Visual interpretation of the 2 metre composited data in 3 dimensions using Gems resulted in determination and iterative subsetting of the data. These subsets were subject to histogram and variogram analysis in order to obtain geostatistical significant grade populations. Refinement of these grade populations to best estimate gold grade stationarity resulted in the determination of numerous mineralogical domains within the broader mineralized zone of each deposit. Slight modifications to existing geological solids and creation of new geological solids from wireframes based on these domains permitted spatial constraints on the data for subsequent interpolation.

Table 14-27: Summary Descriptive Statistics for Raw Assay and 2 m Uncapped Composite Samples

	Minimum	Maximum	Mean	Std. Dev.	Median
NS Raw Assays	0.00	20.17	0.13	0.60	0.013
NS 2m Composites	0.00	15.82	0.13	0.56	0.01
SL Raw Assays	0.00	43.00	0.29	1.02	0.021
SL 2m Composites	0.00	21.54	0.25	0.78	0.02

14.4.7 Estimation

Ordinary Kriging (OK) interpolation methodology was selected to for the Sleeman and North Slope deposits based in the high density and volume available in Golden Predator's drillhole database. Raw and 2 metre composite values were subjected to visual and statistical domaining. A total of 5 domains were defined by variogram analysis for these deposits. Table 14-28 below summarizes the variogram orientation and structures for each domain. Search ellipse parameters were set to the variogram orientation and range. A summary of the search ellipse parameters are listed Table 14-29 below. Orientations reported below are based on the Gems "principal azimuth-principal dip-intermediate azimuth" system. Through iterative model runs, followed by visual inspection, a minimum of 3 to a maximum of 30 composites were required for a value to be assigned to a block. A limit of 6 composites per drillhole was applied. Kriging neighborhood analysis was not performed.

Gems was also used to complete the geostatistical analysis, geological modeling and block modeling for the Mineral Resource estimation. High grade capping was applied to the composited datasets to eliminate positive skew and remove values that were considered anomalous. A high grade cap of 15 g/t was applied to the North Slope and of 16 g/t to the Sleeman data.

A representative cross section of the estimated block model for North Slope is shown in Figure 14–45, and Sleeman in Figure 14–46.

Table 14-28: Summary of Variogram Parameters

Resource Area	Domain	Variogram	P-Azi	P-Dip	Int-Azi	C0	Sill	S-Total
North Slope	11	NS_AU01X	267.621	12.199	9.553	0.550	1.134	1.684
	12	NS_AU01	267.621	12.199	9.553	0.550	1.134	1.684
Sleeman	13	SL_AU55	294.577	8.901	37.037	0.208	2.720	2.928
	14	SL_HG2	302.552	38.866	46.778	0.000	1.431	1.431
	15	SL_HG2	302.552	38.866	46.778	0.000	1.431	1.431

Table 14-29: Summary of Search Ellipse Parameters

Resource Area	Domain	Interpolation Profile	Ellipse	Primary Azimuth	Primary Dip	Int-Azimuth	Major	Semi-major	Minor
North Slope	11	NS_SDNX2	NS_AU01X	267.621	12.199	9.553	35	12	12
		NS_SDOX2							
	12	NS_SEDNX	NS_AU01	267.621	12.199	9.553	69	43	23
		NS_SEDOX							
Sleeman	13	SL_HG1NX	SL_AU55	294.577	8.901	37.037	34	30	14
		SL_HG1OX							
	14	SL_HG2NX	SL_HG2	302.552	38.866	46.778	26	23	12
		SL_HG2OX							
	15	SL_AQM	SL_HG2	302.552	38.866	46.778	26	23	12
		SL_LAQM							
		SL_SED							

Figure 14–45: Oblique Section of North Slope Gold Grade Model (40 metre wide)

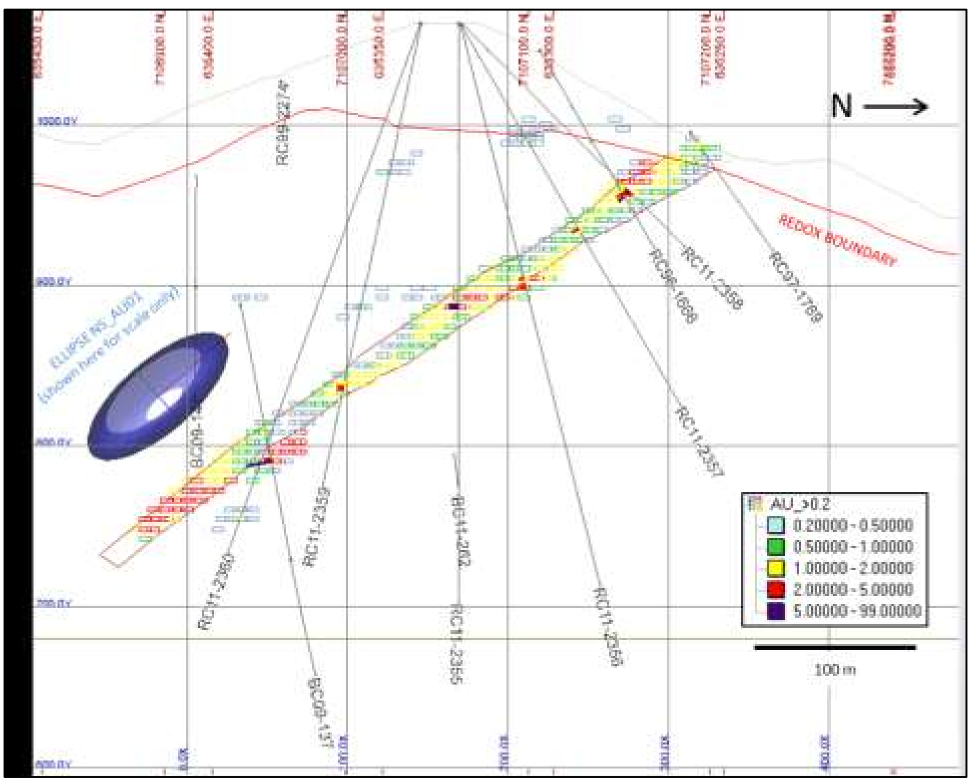
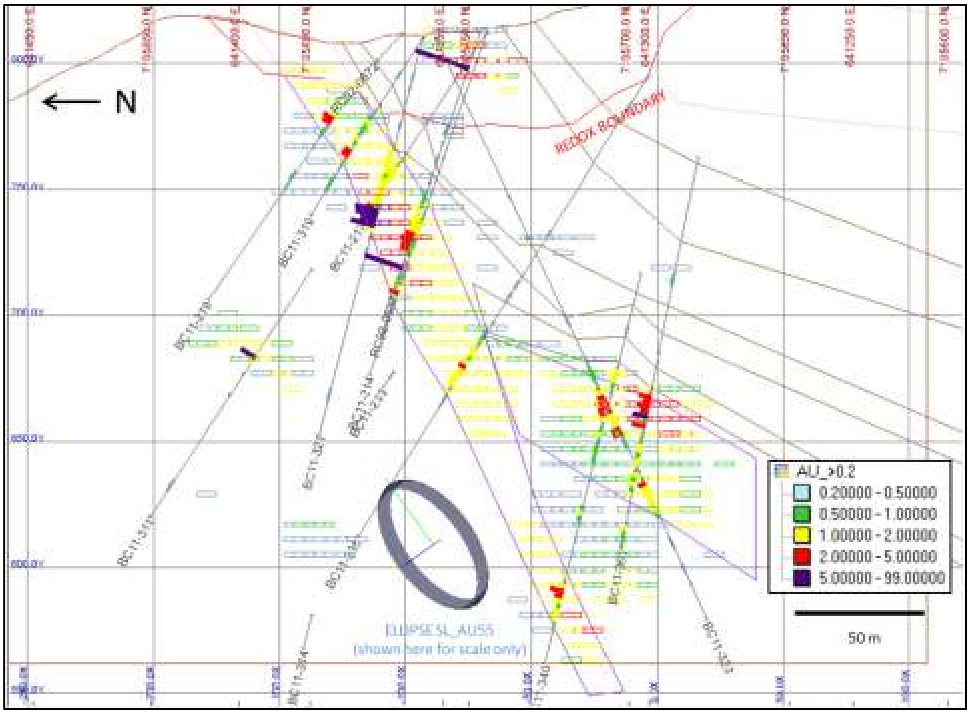


Figure 14–46: Oblique Section of Sleeman Gold Grade Model (25 metre wide)



*Note: Vertical block dimensions may appear less than the actual 6x6x6 metre block size due to the GEMS percent model cross sectional visual representation. Source: EBA (2013)

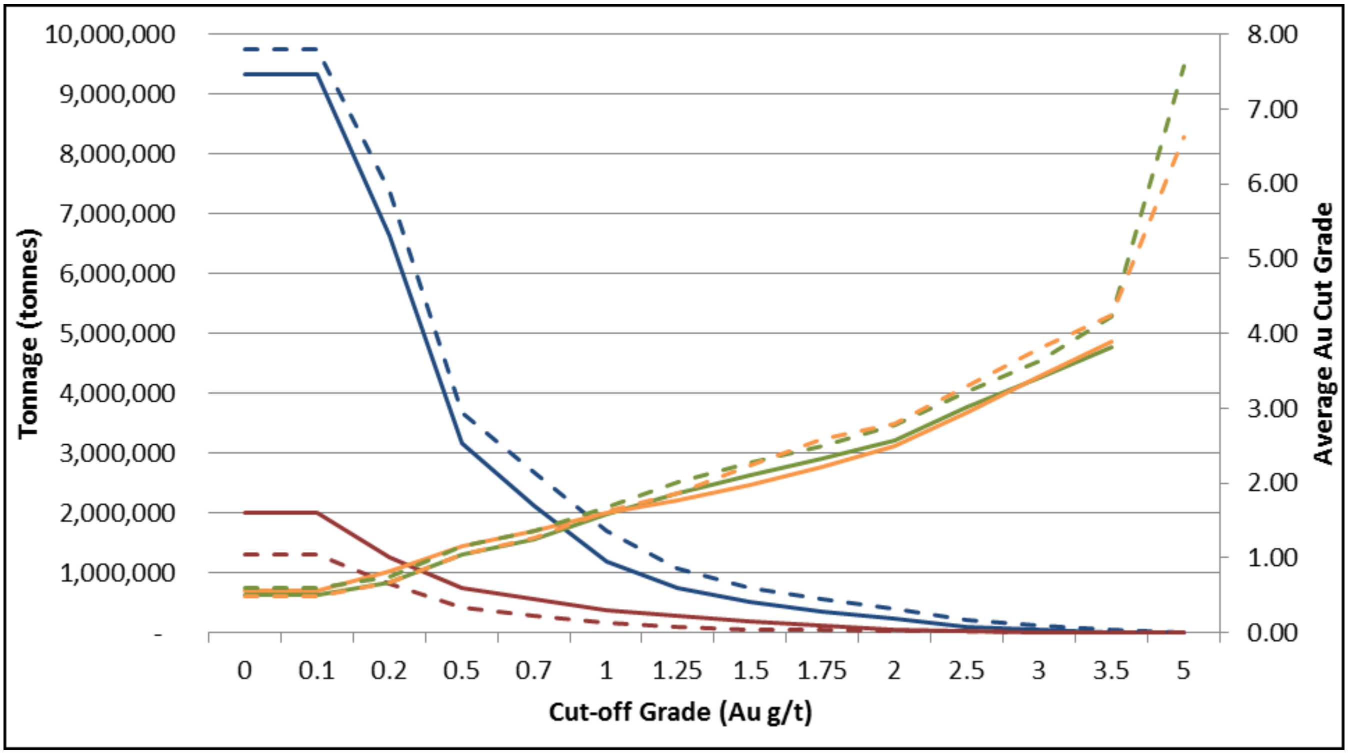
14.4.7.1 Grade Cut-off Selection

Gold cut-off grades of 0.5 g/t Au for oxide material and 0.7 g/t Au for sulfide material have been applied by Tetra Tech EBA for the purposes of this updated mineral resource estimate. The oxide cut-off grade has been revised from 0.2 g/t Au as used in the previous Technical Report to 0.5 g/t Au reported in the previous Technical Report's sensitivity tables to better reflect current market conditions.

The cut-off grades were originally selected based on review NI 43-101 documents reporting on properties analogous to Brewery Creek and since no economic or engineering studies had been completed for the property at the time of the previous report. Tetra Tech EBA feels that the numbers are suitable at the resource estimation stage of the project given a moderate level of uncertainty in the actual costs for the proposed project at this time.

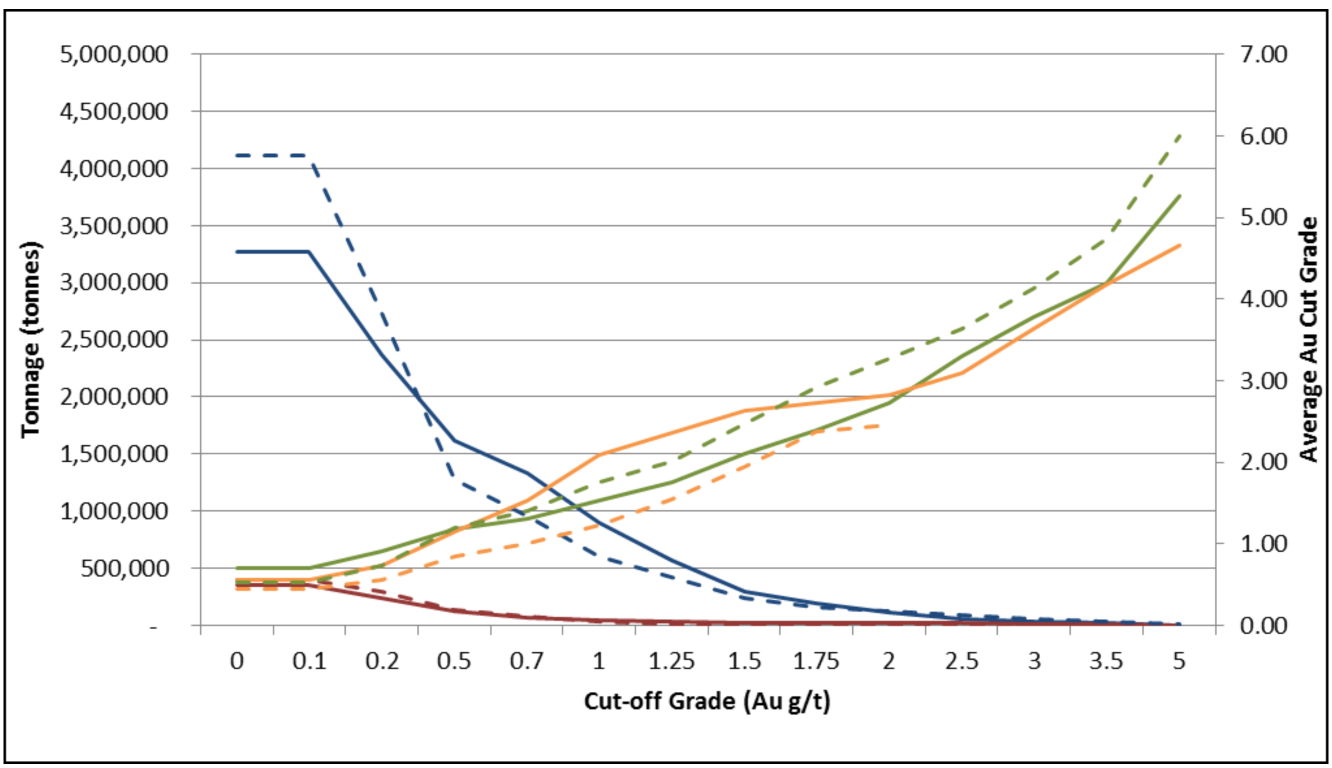
Grade-tonnage curves are presented in Figure 14–47 and Figure 14–48 to highlight the effect with variation to the grade cut-offs. A legend for the line types used in the figures is seen in Figure 14–49.

Figure 14–47: Grade Tonnage Curve for the North Slope Deposit



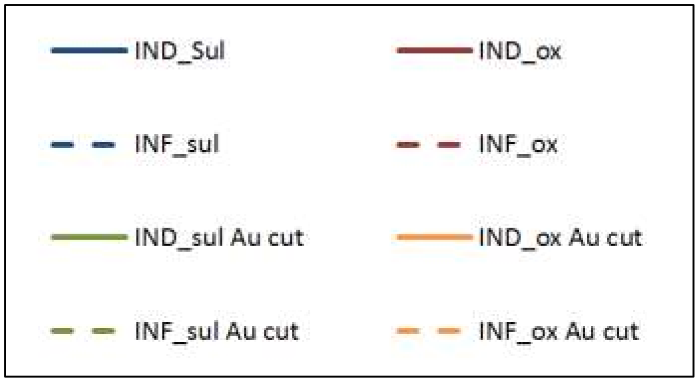
Source: EBA (2013)

Figure 14–48: Grade Tonnage Curve for the Sleeman Deposit



Source: EBA (2013)

Figure 14–49: Legend for grade-tonnage curve line types



(IND = Indicated Resources, IND = Inferred Resources, ox = oxide material, sul = sulfide bearing material, cut = gold grade subject to high grade cap)
Source: EBA (2013)

14.4.7.2 Model Validation

Three validation methods were conducted to compare the results of the six individual block models as a reasonable estimate of the raw data: visual comparison of raw assays to the block model, geostatistical comparison of raw and 2 metre composites to the block model and unbiased nearest neighbor interpolation method compared to the Kriging/IDW interpolation methods. The block model data was exported from Gems as only the blocks that were candidate for either Inferred or Indicated Resource Classification, by this only blocks with grades greater or equal to 0.1 g/t Au were included in the analysis.

14.4.7.3 Visual Comparison

Visual comparison of raw assay and 2 metre composited data plotted on drillholes versus the gold distribution within the block model was completed. It was felt that a good correlation between the data was seen and that no significant biases were apparent in the block model data.

14.4.7.4 Geostatistical Comparison

A summary of the descriptive statistics for gold values within the block models were compared to the data set for the raw assays and the 2 metre composites. Table 14-30 below summarizes the comparative statistics where all raw and 2 metre composite data less than 0.1 g/t gold were removed from the population to be comparable with the block model data.

Block model data is reported here with lower mean and median values. This is felt to be justified by inherent declustering of raw and 2 metre composite sampling within the block model, for which areas with numerous drillholes in the same high grade zone have been declustered. In addition, 2 metre composite values of less than 0.1 g/t gold may have contributed to lower average grades of some marginal blocks, as these same less than 0.1 g/t gold values have been excluded from the raw and 2 metre composite datasets a minor low grade bias is influencing the block model data.

Table 14-30: Resource Block Model Comparative Statistics

Resource Area	Resource Block Model			Raw			2m Composite		
	Mean	Median	Std. Dev	Mean	Median	Std. Dev	Mean	Median	Std. Dev
North Slope	0.550	0.337	0.570	0.737	0.298	1.359	0.693	0.290	1.226
Sleeman	0.606	0.352	0.655	0.944	0.505	1.730	0.817	0.450	1.295

14.4.7.5 Nearest Neighbor Comparison

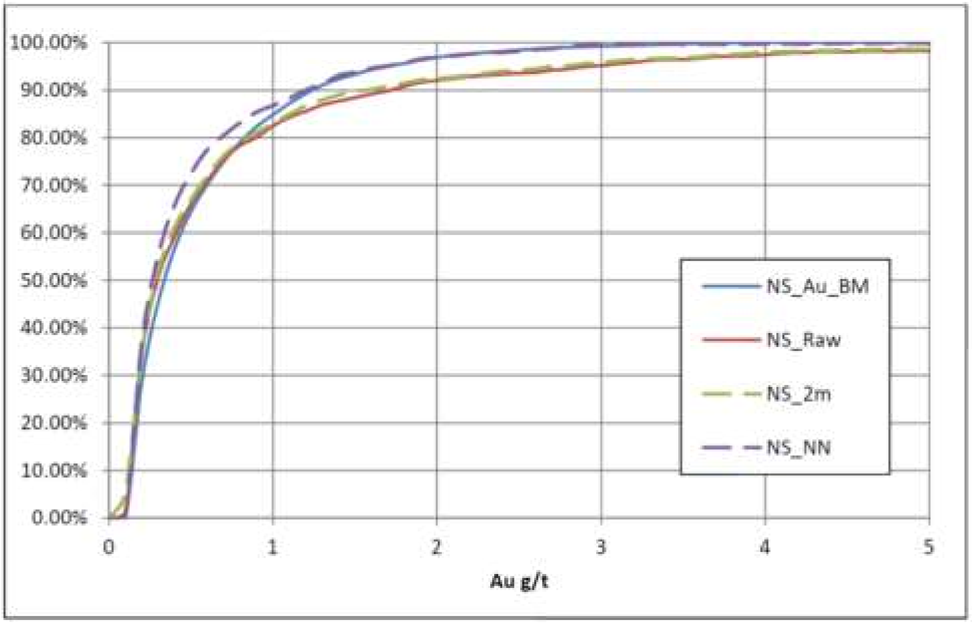
An exercise was completed where the complete 2 metre composite datasets for each deposit were interpolated into a validation block model using an ‘unbiased’ nearest neighbor (NN) model. This method does not factor distance, direction or clustering into account when determining block grades during the interpolation process and is intended to provide a ‘raw’ numerical representation of an interpolated grade distribution. This method does not incorporate extensive control parameters, or ‘bias’, to the interpolation profile and typically does not visually represent grade trends or mineral continuity well.

An isotropic ellipse with a 30 metre radius was used to compare NN models with the 5 Kriged deposits and an isotropic ellipse with radius of 50 metres was used to compare the NN model with the Classic Deposit (interpolated using IDW). The radius was selection to reflect a generalized average of the search range determined for each deposit from the variography.

Given the consistent and relatively dense drillhole spacing within the mineralized zones, the NN model resulted visually in a remarkably similar grade distribution as the Kriging and IDW models, however, as no geological control was implemented on the NN models, grade was noted to cross unmineralized geological boundaries.

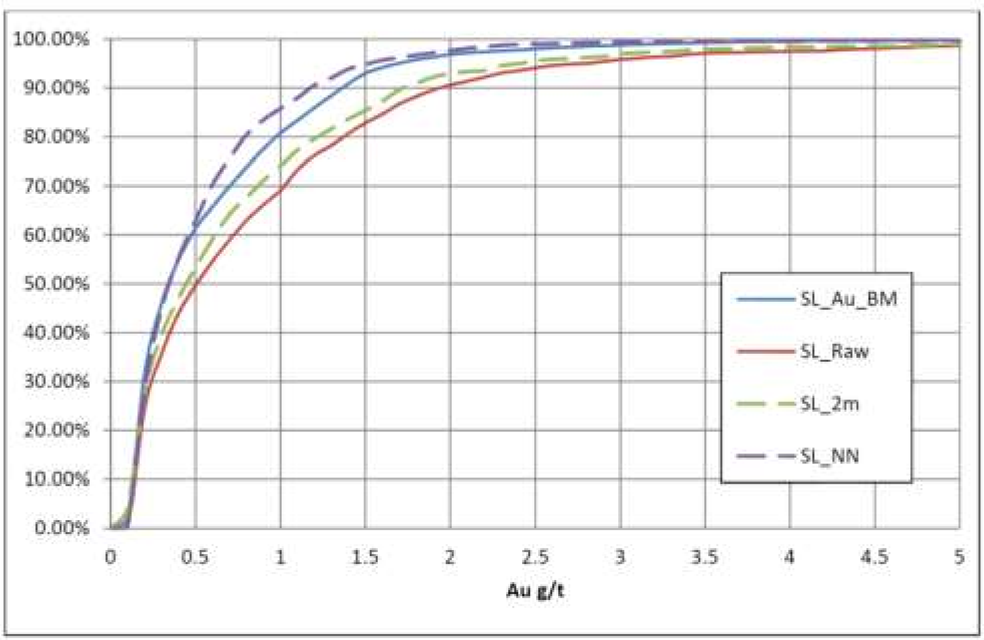
Cumulative probability plots were created to compare grade populations for the raw, 2 metre composite, resource block model and NN block model data, where all data sets were filtered of data less than 0.1 g/t Au. The plots are presented as Figure 14–50 and Figure 14–51, below. Review of the plots reveals that the resource block models and NN models are numerically similar. Some smoothing is noted within both block model datasets relative to the raw dataset, however, the resource block model data and the 2 metre composite data are typically found to lie between the two extremes of the NN model and the raw dataset. As discussed above, declustering of the source data and the inherent low grades within the marginal blocks may contribute to some of the smoothing effect present in the Big Rock, Fosters-Canadian and Bohemian Schooner Deposits.

Figure 14–50: Cumulative Probability Plot for North Slope Deposit



Source: EBA (2013))

Figure 14–51: Cumulative Probability Plot for Sleeman Deposit



Source: EBA (2013)

14.5 Mineral Resource Classification

The mineral resources estimates reported here have been classified as either Indicated or Inferred, as defined in the CIM Definition Standards on Mineral Resources and mineral Reserves. Classification schemes are included below by deposit.

This Preliminary Economic Assessment includes both Inferred and Indicated mineral resources for the East Big Rock, West Big Rock, Lower Fosters, Kokanee, Golden, Lucky, Bohemian, Schooner and the historical leach pad. Inferred mineral resources are considered to be based on some geological speculation and do not have the inherent confidence that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment of these deposits will be realized.

14.5.1 Kokanee, Golden, Pacific, Blue, and Lucky Deposits

The estimated block grades were classified into Indicated and Inferred categories. For the Kokanee, Golden, Pacific, Blue, and Lucky deposits oxide blocks that were estimated by two or more drillholes with the closest hole within 25m were classified as Indicated Resources. Also, blocks in KOGD and PABL that were estimated by a single drillhole within 15 metres of the nearest drillhole, and blocks in LU that were estimated by a single drillhole within 7.5 metres of the nearest drillhole were classified as Indicated Resources. All other estimated oxide blocks were classified as Inferred. Oxide blocks were reported inside a $1250 Au LG pit. The parameters for pit construction are listed in Table 14-31. These pit parameters were used to construct a base case Au cut-off grade for resource reporting, as noted in Equation 3. All sulfide blocks were classified as Inferred. No Measured Mineral Resources have been defined within the current Mineral Resource Estimate.

14.5.2 Bohemian, Schooner, Lower Fosters, West Big Rock, East Big Rock, Classic, and Lone Star Deposits

The estimated block grades were classified into Indicated and Inferred categories. For the Bohemian, Schooner, Lower Fosters, West Big Rock, and East Big Rock deposits oxide blocks that were estimated by two or more drillholes with the closest hole within 25m were classified as Indicated Resources. Blocks estimated by a single drillhole within 12.5m were also classified as Indicated Resources. All other estimated oxide blocks were classified as Inferred. Oxide blocks in the above areas were reported inside a $1250 Au LG pit. The parameters for pit construction are listed in Table 14-31. These pit parameters were used to construct a base case Au cut-off grade for resource reporting, as noted in Equation 3. All sulfide blocks were classified as Inferred. Blocks were considered as Inferred Resources for the Classic and Lone Star deposits if they were estimated by three or more drillholes with the closest hole located within 50 metres or two or more holes with the closest within 37.5 metres or 1 hole within 25 metres. No Measured Mineral Resources have been defined within the current Mineral Resource Estimate.

14.5.3 Historical Heap Leach Pad

Indicated Resources were confined to the base of the sampled sonic drillholes upward to the existing heap surface. All material below the base of the assayed sonic holes to a surface located 3 metres above the heap leach liner was classified as Inferred Resources. The lower portion of the existing heap leach pad was placed into the Inferred category primarily due to a lack of sonic drillhole assays from the lower levels of the pad, and to provide a conservative buffer zone to protect the liner from being breached. The base case Au cut-off was calculated based on assuming a $1250/oz Au price, and a 45% recovery.

14.5.4 North Slope and Sleeman Deposits

An Inferred classification has been applied to target marginal and outlier blocks in the block models, which suggest presence and continuity of mineralization but lack the density of data for confirmation. Inferred blocks lie within the maximum variogram range, are associated with one or more drillholes, exist on the outer extremities of the principal mineralized body, have at least 3 composite samples reporting within the search ellipse and contain partial block grades of greater than or equal to 0.1 g/t Au. An Indicated classification has been applied to block models to target portions of the mineralized body where data density confirms the presence and continuity of mineralization with a moderate level of confidence. Indicated blocks lie within 25 metres of the closest reporting composite, have a minimum of 8 composites reporting within the search ellipse from a minimum of 2 drillholes and contain partial block grades of greater than or equal to 0.1 g/t Au. These minimum criteria were selected based on visual inspection of the grade distribution and the average drillhole spacing. No Measured Mineral Resources have been defined within the current Mineral Resource Estimate.

In early 2012, when the resource estimates for Sleeman and North Slope were calculated no economic or engineering studies had been completed for the property and gold cut-off grades of 0.5 g/t Au for oxide material and 0.7 g/t Au for sulfide material were selected based reviews of NI 43-101 documents reporting on other properties analogous to Brewery Creek and based on the gold prices at the time. Tetra Tech EBA felt that these numbers were suitable for the resource estimation stage of the project.

14.6 Mineral Resource Estimation

Oxide Mineral Resources for the Kokanee, Golden, Pacific, Blue, Lucky, Bohemian, Schooner, Lower Fosters, West Big Rock, and East Big Rock deposits are based on Indicated and Inferred Resources which are located inside of conceptual pits. Table 14-31 summarizes the parameters that were used to generate conceptual oxide pit shells (no value was attributed to sulfide material). Both Indicated and Inferred Resources were used to generate the pit shells. Oxide cut-off grades for resource declaration were established using the parameters shown in Table 14-31.

Table 14-31: Oxide Pit Parameters

Resource Area	Costs					Au Cut-off (g/t)*
	Mining ($/tonne)	G&A ($/tonne leached)	Processing ($/tonne leached)	LG Processing ($/tonne leached)	Au Recovery	
Kokanee, Golden	$3.10	$2.65	$9.41	$12.06	70%	0.54
Pacific, Blue	$2.78	$2.65	$9.41	$12.06	70%	0.53
Lucky	$3.20	$2.65	$9.41	$12.06	70%	0.54
Bohemian	$3.20	$2.65	$9.41	$12.06	77%	0.49
Schooner	$3.20	$2.65	$9.41	$12.06	74%	0.51
Lower Fosters	$2.97	$2.65	$9.41	$12.06	73%	0.51
West Big Rock	$2.92	$2.65	$9.41	$12.06	83%	0.45
East Big Rock	$2.92	$ 2.65	$9.41	$12.06	77%	0.48

Equation 2: Basecase Au Cut-off Calculation

$$Au\ Basecase\ Cutoff = \frac{Mining\ Cost + Processing\ Cost + G\&A\ Cost}{Au\ Price * Au\ Recovery}$$

All mineral resource estimates are summarized in Table 14-32 and Table 14-33. Only oxide resources from the West and East Big Rock, Lower Fosters, Kokanee, Golden, Lucky, Bohemian, Schooner and Historical Leach Pad areas are considered in the PEA. All sulphide resources and all resources from the Classic, Lonestar, Sleeman and North Slope have been omitted from the PEA.

Mineral resources are not mineral reserves and do not demonstrate economic viability. The quantity and grade of inferred resources reported herein are uncertain in nature and exploration completed to date is insufficient to define these Mineral Resources as indicated or measured. There is no guarantee that further exploration will result in the inferred Mineral Resources being upgraded to an indicated or measured mineral resource category. There is no certainty that all or any part of the mineral resource will be converted to mineral reserves. Mineral Resources are not mineral reserves and may be materially affected by environmental, permitting, legal, socio-economic, marketing, political, or other factors. Tetra Tech is not currently aware of information which may affect the Mineral Resources. Quantity and grade are estimates and are rounded to reflect the fact that the resource estimate is an approximation. The Mineral Resources have an effective date of June 1, 2013.

Table 14-32: Summary of Total Mineral Resources – Oxide Mineral Resources

Resource Area	Au Cut-off (g/t)	Indicated Oxide Resources			Inferred Oxide Resources			Constrained by $1250 LG Pit?	QP Responsible	Estimated with Capped Composites?
		Tonnes (000)	Au (g/t)	Au Ozs (000)	Tonnes (000)	Au (g/t)	Au Ozs (000)			
Kokanee	0.54	1,201	1.19	46	279	1.19	11	Yes	Don Hulse	Yes
Golden	0.54	1,070	1.38	47	247	1.25	10	Yes	Don Hulse	Yes
Pacific	0.53	373	1.01	12	131	0.91	4	Yes	Don Hulse	Yes
Blue	0.53	250	1.29	10	29	0.98	1	Yes	Don Hulse	Yes
Lucky	0.54	2,394	1.36	105	236	1.27	10	Yes	Don Hulse	Yes
Bohemian	0.49	1,491	1.31	63	134	1.49	6	Yes	Mike Lechner	Yes
Schooner	0.51	1,108	1.99	71	243	2.65	21	Yes	Mike Lechner	Yes
Lower Fosters	0.51	1,090	1.61	56	492	1.52	24	Yes	Mike Lechner	Yes
West Big Rock	0.45	722	1.27	29	38	0.75	1	Yes	Mike Lechner	Yes
East Big Rock	0.48	596	1.10	21	21	0.87	1	Yes	Mike Lechner	Yes
Classic	0.54	-	-	-	3,711	0.81	97	No	Mike Lechner	Yes
Lone Star	0.54	-	-	-	1,522	0.88	43	No	Mike Lechner	Yes
North Slope	0.5	756	1.15	28	412	1.05	14	No	James Barr	Yes
Sleeman	0.5	124	1.14	5	132	0.84	4	No	James Barr	Yes
Historical Viceroy Pad	0.30	2,977	0.88	84	1,682	0.60	32	No	Mike Lechner	No
Total		**14,152**	**1.27**	**577**	**9,309**	**0.93**	**279**			

Table 14-33: Summary of Total Mineral Resources – Sulfide Mineral Resources

Resource Area	Au Cut-off (g/t)	Indicated Sulfide Resources			Inferred Sulfide Resources			Constrained by $1250 LG Pit?	QP Responsible	Estimated with Capped Composites?
		Tonnes (000)	Au (g/t)	Au Ozs (000)	Tonnes (000)	Au (g/t)	Au Ozs (000)			
Kokanee	0.70	-	-	-	1,547	1.33	66	No	Don Hulse	Yes
Golden	0.70	-	-	-	649	1.20	25	No	Don Hulse	Yes
Pacific	0.70	-	-	-	707	1.45	33	No	Don Hulse	Yes
Blue	0.70	-	-	-	1,358	1.31	57	No	Don Hulse	Yes
Lucky	0.70	-	-	-	1,783	1.36	78	No	Don Hulse	Yes
Bohemian	0.70	-	-	-	973	1.58	50	No	Mike Lechner	Yes
Schooner	0.70	-	-	-	313	1.42	14	No	Mike Lechner	Yes
Lower Fosters	0.70	-	-	-	883	1.45	41	No	Mike Lechner	Yes
West Big Rock	0.70	-	-	-	381	1.28	16	No	Mike Lechner	Yes
East Big Rock	0.70	-	-	-	170	1.00	5	No	Mike Lechner	Yes
Classic	0.70	-	-	-	-	-	-	No	Mike Lechner	Yes
Lone Star	0.70	-	-	-	-	-	-	No	Mike Lechner	Yes
North Slope	0.70	2,122	1.26	86	2,686	1.36	118	No	James Barr	Yes
Sleeman	0.70	1,337	1.30	56	958	1.40	43	No	James Barr	Yes
Total		**3,459**	**1.28**	**142**	**12,408**	**1.37**	**546**			

15.0 MINERAL RESERVE ESTIMATES

This PEA is preliminary in nature and provides no certainty that the mine plan or the projected cash flows will be realized. There are currently no mineral reserves defined for the property.

16.0 MINING METHODS

16.1 Introduction

Mining plans, mining methods and mine schedules have been produced for 8 of the deposits within the Brewery Creek property for the purpose of the PEA. These are from west to east, West Big Rock, East Big Rock, Fosters, Kokanee, Golden, Lucky, Bohemian and Schooner. Mining plans and processing methods are currently restricted to oxide inferred and indicated resources.

Due to the shallow nature of the resources included in the PEA, only open pit mining has been considered. The mining has been proposed as a truck and shovel type operation. Where existing pits occur, these would be expanded and deepened as the case may be and existing haul roads will be restored for use to access the historic and new deposit locations. The mine planning process involved the use of Geovia Whittle 4.5™ (Whittle) software, which uses the Lerch-Grossman (LG) algorithm, to create theoretical optimized pit shells. The LG shells have been imported into GeoVia Gems™ (Gems), for creation of open pit designs, which include benched pit slopes and haul roads.

Since multiple deposits form part of this PEA, a multi-pit module of the Whittle software was used to create a strategic mining schedule. No tactical scheduling work has been done for the deposits.

16.2 Lerch-Grossman Optimization

The optimization work was undertaken using parameters derived from initial mining investigations into pit slopes, mining and processing costs and current gold price ranges.

Parameters that were common to all deposits are listed below.

1. Gold price in CAD$ per troy ounce CAD $1,250 / troy oz.
2. Waste mining cost CAD$2.82 / tonne
3. Mining dilution 5%
4. Mining losses 5%
5. Gold selling costs CAD $ 32 / troy oz.

Pit slopes were applied as 48 degrees from horizontal, in accordance with pit slope design results in section 16.3.

Recoveries and process costs were applied as shown in Table 16-1. Two costs are entered into Whittle to undertake scheduling, namely the mining cost and the processing cost. The mining cost was input as $2.82 for all rock mined with the processing cost being the additional cost applicable to mining, processing and general and administrative costs for rock selected for processing. No additional costs were included for depth, as costs were applied as average costs for mining the entire pit.

Table 16-1: Process Costs and Recoveries Used in Pit Optimization

Ore Body	Additional Cost Per Tonne Processed ($/tonne processed)	Au Recovery
East Big Rock	$12.16	77%
West Big Rock	$12.16	83%
Bohemian	$12.44	77%
Schooner	$12.44	74%
Fosters	$12.21	73%
Kokanee and Golden	$12.34	70%
Lucky	$12.44	70%

16.3 Pit Slope Design

The geotechnical data for the assessment of the proposed pits was obtained from site diamond drilling investigation program in July and August of 2012. Surface structural mapping of selected benches of historical pits (Moosehead, Canadian, and Fosters) and road cuts located in the Big Rock West, Bohemian and Schooner proposed pit areas was conducted during September 2012, to complement the drilling investigation. In addition to the geotechnical drilling program and mapping, the study included a thorough review of previous background reports and geological maps that were considered pertinent to the project. No geotechnical drilling or analysis was done for Kokanee, Golden and Lucky and as such the results for the other deposit areas have been applied to these proposed pits.

This section summarizes the steps conducted for assessing the stability of the pit walls including:

- Geotechnical and structural domain models;
- Rock Mass Classification;
- Rock mass structures;
- Slope stability assessment; and
- Recommended slope configuration.

16.3.1 Rock Mass Classification

Using the 2012 drilling and mapping data, Tetra Tech EBA assessed the rock mass quality of the pit areas using the Rock Mass Rating (RMR) system (Bieniawski 1976). Table 16-2 details the range of representative RMR values (Bieniawski, 1976) at five of the proposed pit sites, Kokanee, Lucky and Golden have not been included in the geotechnical study for the PEA.

Table 16-2: Results of RMR Analysis

Pit	Percentage of Representative RMR Value Range					
	< 30	30 to 40	40 to 50	50 to 60	60 to 70	> 70
West Big Rock	25.6	24.5	24.9	14.3	7.7	2.9
East Big Rock	20.4	23.3	24.0	22.2	7.2	2.9
Lower Fosters	18.9	28.4	32.4	13.5	5.4	1.4
Bohemian	11.8	8.2	10.2	25.5	20.4	23.9
Schooner	31.1	20.0	22.2	20.0	5.9	0.7

16.3.2 Geotechnical and Structural Domains

The geotechnical model is composed of individual regions (domains) each of which comprise materials with similar geotechnical rock mass properties, and similar geological and structural characteristics. Based on the available geotechnical data (geological rock types and the geotechnical rock mass properties), Tetra Tech EBA developed a simplified geotechnical model by separating the rock mass into three main geotechnical domains (D1, D2, and D3). Domain D1 corresponds to volcanic tuff, D2 to intrusive monzonite, and D3 to sedimentary rocks.

In general, the lower part of the hanging wall of the pits consist of Domain D2 material, and the upper part of Domain D3. The footwall will consist of either Domain D3 material if all the monzonite is excavated or D2 material if part of the monzonite is not excavated.

16.3.3 Rock Mass Structures

Rock structures in the area shows two major classes: rock fabric and major structures. This differentiation relates largely to the continuity of the features and the resultant impact with respect to the slope design elements.

Rock fabric data was obtained for this PEA from the 2012 geotechnical investigations (oriented boreholes and surface mapping). Stereographic analyses were conducted on each of the pits with the collected fabric data. The structural evaluation is undertaken mainly to assess the stability of the bench faces and in some degrees multiple bench stability. Table 16-3 presents the joint sets, which have formed the basis of the kinematic analysis for the Brewery Creek pits, excluding Kokanee, Golden and Lucky.

Table 16-3: Joint Sets for the Identified Domains

Pit	Domain	Joint Set ID (Dip/Dip Direction)
West Big Rock	D 2	J1 (33/173), J2 (83/233), J3 (55/075)
	D 3	J1 (56/192), J2 (59/290), J3 (90/331), J4 (46/112)
East Big Rock	D 2	J1 (33/173), J2 (83/233), J3 (55/075)
	D 3	J1 (56/192), J2 (59/290), J3 (90/331), J4 (46/112)
Lower Foster	D 2	J1A (73/175), J1B (53/166), J2 (20/251), J3 (87/114), J4A (74/041), J4B (54/071), J5 (84/334)
	D 3	J1(88/142), J2 (38/151), J3 (65/242), J4 (78/108), J5 (10/016), J6A (90/043), J6B (80/023)
Bohemian	D 2	J1A (70/207), J1B (46/218), J2A (56/080), J2B (73/100), J3 (77/139), J4 (79/173), J5 (57/340)
	D 3	J1 (07/171), J2 (40/003), J3 (87/109), J4 (85/176), J5 (51/173)
Schooner	D 2	J1A (70/207), J1B (46/218), J2A (56/080), J2B (73/100), J3 (77/139), J4 (79/173), J5 (57/340)
	D 3	J1 (07/171), J2 (40/003), J3 (87/109), J4 (85/176), J5 (51/173)

Major structures consist of relatively continuous features, such as folds and inter-ramp and site scale faults, capable of contributing to large scale instability (multiple bench failure and overall slope pit wall). Thrust faults at Brewery Creek generally strike east-northeast (±070° azimuth), dip moderately southeast, and commonly place siltstone of the Steele formation above variably graphitic and locally baritic argillite of the Earn group. At least three orientations of high-angle faults occur at Brewery Creek: one set strikes northeast, another strikes northwest, and the other east-northeast; all are steeply dipping.

16.3.4 Rock Slope Stability Assessment

The RMR values at the proposed pit areas in the Brewery Creek Property range mainly between 25 and 50. Therefore, the controlling failure could be either structural or non-structural (i.e., based on rock mass strength), but is more likely to involve a complex mixture of both. The following slope stability analyses were performed for each pit:

- Structurally controlled failures - Kinematic analysis; and
- Rock mass strength controlled failures - Limit equilibrium methods.

The pit slope angles for each pit were selected as the shallower slopes between the angles resulting from the analyses described in the two bullets above.

16.3.5 Pit Slope Assessment – Kinematic Analysis

Kinematic analysis of the structural geology (rock fabric) was used to search for potential failures controlled by adverse structural conditions in relation to the direction of the bench faces. These may take the form of planar failures on outward dipping discontinuities, wedge failures on intersecting discontinuities, toppling on inward dipping discontinuities or complex failures modes involving all of these processes.

Bench face angles (BFA), where rock structure is the primary failure control, depend on the wall orientation. Therefore, the pit slopes were evaluated for such regions of similar structural characteristics and pit slope orientations in “design sectors” which are expected to exhibit similar response to pit development.

The rock mass within open pit benches is usually moderately disturbed by blasting, and the pits will be developed on ridges where groundwater is low. Therefore, no pore water pressures were considered in the bench scale stability analysis. In addition, due to the blast damage and the relative low stress involved, the cohesion of the joint plane discontinuities was ignored in the analysis. For design purposes, a residual friction angle of 30°, 32° and 35° (for rock to rock contact) was adopted for the discontinuities of sedimentary, intrusive monzonite and volcanic tuff rocks, respectively. Table 16-4 summarizes the results of the kinematic stability analysis for planar and wedge analyses, excluding Kokanee, Golden and Lucky. The lower of the two governs the kinematic stability of the bench.

Table 16-4: Summary of Kinematic Bench Face Instability Analyses

Pit	Sector	Pit Wall Dip Direction	Sector Azimuth	Domain	Potential Mode of Failure		Optimum BFA (°)	Adopted Design BFA[2] (°)	Bench Height (m)	Catch Bench Width (m)	IRA[3] (°)	Risk[4]
					Planar[1]	Wedge[1]						
West Big Rock	1	154	334	D 2	33° (74.4°)	90° (90°)	74.4	65	18	8	48	Moderate
				D 3	90° (90°)	43° (76.3°)	76.3	65	18	8	48	
	2	184	4	D 2	33° (74.4°)	90° (90°)	74.4	65	18	8	48	Low
				D 3	56° (79.4°)	52° (78.4°)	78.4	65	18	8	48	
	3	337	157	D 2	90° (90°)	90° (90°)	90	65	18	8	48	Low
				D 3	90° (90°)	90° (90°)	90	65	18	8	48	
East Big Rock	1	162	342	D 2	33° (74.4°)	90° (90°)	74.4	65	18	8	48	Low
				D 3	90° (90°)	45° (76.7°)	76.7	65	18	8	48	
	2	182	002	D 2	33° (74.4°)	90° (90°)	74.4	65	18	8	48	Low
				D 3	56° (79.4°)	51° (78.2°)	78.2	65	18	8	48	
	3	000	180	D 2	90° (90°)	90° (90°)	90.0	65	18	8	48	Low
				D 3	90° (90°)	90° (90°)	90.0	65	18	8	48	
Fosters	1	179	359	D 2	53° (78.7°)	50° (77.9°)	77.9	65	18	8	48	Moderate
				D 3	90° (90°)	36° (77.3°)	77.3	65	18	8	48	
	2	269	089	D 2	90° (90°)	71° (83.8°)	83.8	65	18	8	48	Low
				D 3	90° (90°)	70° (84.6°)	84.6	65	18	8	48	
	3	000	180	D 2	90° (90°)	67° (82.6°)	82.6	65	18	8	48	Low
				D 3	90° (90°)	80° (87.2°)	87.2	65	18	8	48	
	4	088	268	D 2	54° (78.9°)	47° (77.2°)	77.2	65	18	8	48	Low
				D 3	78° (86.7°)	46° (79.1°)	79.1	65	18	8	48	
Bohemian	1	132	312	D 2	77° (85.7°)	59° (80.3°)	80.3	65	18	8	48	Low
				D 3	90° (90°)	68° (84.1°)	84.1	65	18	8	48	
	2	180	360	D 2	79° (86.4°)	51° (78.2°)	78.2	65	18	8	48	
				D 3	51° (80.1°)	50° (79.9°)	79.9	65	18	8	48	
	3	223	043	D 2	46° (77.0°)	46° (77.0°)	77.0	65	18	8	48	Low
				D 3	90° (90°)	52° (80.3°)	80.3	65	18	8	48	
	4	296	116	D 2	90° (90°)	79° (86.4°)	86.4	65	18	8	48	Low
				D 3	90° (90°)	84° (88.3°)	88.3	65	18	8	48	
	5	017	197	D 2	90° (90°)	45° (76.7°)	76.7	65	18	8	48	Moderate
				D 3	40° (78.0°)	39° (77.8°)	77.8	65	18	8	48	
Sch oon	1	180	360	D 2	79° (86.4°)	52° (78.4°)	78.4	65	18	8	48	Moderate

Table 16-4: Summary of Kinematic Bench Face Instability Analyses

Pit	Sector	Pit Wall Dip Direction	Sector Azimuth	Domain	Potential Mode of Failure		Optimum BFA (°)	Adopted Design BFA[2] (°)	Bench Height (m)	Catch Bench Width (m)	IRA[3] (°)	Risk[4]
					Planar[1]	Wedge[1]						
				D 3	51° (80.1°)	50° (79.9°)	79.9	65	18	8	48	
	2	270	90	D 2	90° (90°)	61° (80.8°)	80.8	65	18	8	48	Low
				D 3	90° (90°)	90° (90°)	90.0	65	18	8	48	
	3	179	179	D 2	58° (80.0°)	49° (77.7°)	77.7	65	18	8	48	Low
				D 3	40° (78.0°)	41° (78.2°)	78.0	65	18	8	48	
	4	268	268	D 2	56° (79.4°)	55° (79.2°)	79.2	65	18	8	48	Low
				D 3	90° (90°)	63° (82.8°)	82.8	65	18	8	48	

[1] The first angle represents the angle of the potential mode of failure for a continuous fabric persistence. The second angle (within parenthesis) represents the angle of the potential mode of failure for a fabric persistence of 6 m.

[2] Maximum BFA was restrained to 65°.

[3] The IRA angles tabulated above correspond to the angle obtained by translating up the bench geometrical configuration (BFA, bench height and bench width).

[4] Risk tabulated above is associated with the probability of failure and the associated impacts (consequences). Low risk is assigned where slope failure is not to compromise safety and impact the proposed operations. High risk is assigned where safety of personnel is potentially compromised and the operations could be affected and which could result in economic loss. Moderate failure is where safety of personnel is not compromised, but the operations may be delayed until the failure is addressed.

16.3.6 Rock Mass Strength Analysis

The likelihood of generating large-scale failure through the rock mass was assessed with a limited equilibrium program, Slope/W for the expected pit heights for each pit. An acceptable level for the factor of safety was set at 1.2 in the static conditions (the expected peak ground acceleration (PGA) at the site was determined to be so inconsequential and, therefore, no seismic loadings were considered in the analysis conducted for this evaluation). Due to the expected irregularity of the permafrost throughout the proposed pits, the pit slope assessment at overall scales was based on unfrozen rock mass conditions. The analyses considered dry conditions for the upper half of the pit wall and wet conditions for the lower half in the West Big Rock, East Big Rock and Lower Fosters Pits. For the Bohemian and Schooner Pits, the analyses considered wet conditions for lower one-third of the pit wall. Table 16-5 summarizes the conducted limit equilibrium results excluding Kokanee, Golden and Lucky.

Table 16-5: Summary of Limit Equilibrium Analyses

Pit	Pit Sector	HW/FW	Slope Height Trial (m)	Overall Angle Trial	Factor of safety
West Big rock	1 & 2	FW	55	53	1.2
			81	45	1.2
	3	HW	65	65	1.6
East Big Rock	1 & 2	FW	58	54	1.3
			69	54	1.2
	3	HW	41	56	2.3
Lower Fosters	1 & 2	FW	137	35	1.2
	3 & 4	HW	36	60	1.9
Bohemian	1, 2 & 3	FW	85	58	1.6

Table 16-5: Summary of Limit Equilibrium Analyses

Pit	Pit Sector	HW/FW	Slope Height Trial (m)	Overall Angle Trial	Factor of safety
	4 & 5	HW	39	56	2.4
Schooner	1 & 2	FW	53	55	1.2
			73	50	1.2
			84	45	1.2
	3 & 4	HW	96	45	1.2

[1] HW: Hanging wall; FW: Footwall; END: refers to the ends of the pit between the HW and FW.
Pit slopes of 48 ° were used for Kokanee, Golden and Lucky.

16.3.7 Recommended Slope Configuration

16.3.7.1 Bench Face Angle and Inter-ramp Angle Recommendations

Table 16-5 summarize the recommended slope configurations for the six proposed pits to approximately the bottom of the oxide zone. The angles presented in the table correspond to the shallower pit slope between the angles resulting from the instabilities analysed by kinematic and rock mass strength.

16.3.7.2 Bench Width Recommendations

For this PEA Stage, the required bench width was estimated using the empirical relationship developed by Call and Nicholas Inc. (Call et al. 2001).

Required bench width (m) = 0.2 x bench height + 4.5 m.

Using this empirical relationship, a bench width of 8 m is recommended for a bench height of 18 m for PEA purposes.

For geotechnical stability analyses of the heap leach facility refer to section 17.1.2.

16.3.8 Open Pit Hydrology

The general groundwater regime at the Brewery Creek Property consists of a bedrock aquifer which is partly overlain and confined by discontinuous permafrost. The bedrock aquifer, in which groundwater flow mainly occurs along fractures and other rock discontinuities, has predominant bedrock lithologies of monzonite and argillite. Groundwater flow in fractured media is complex and can vary greatly in direction and rate, depending on the local hydrogeological and structural geological conditions. Transmissivity values can change over several orders of magnitude within the same rock mass, and groundwater flow may be largely controlled by a few conductive fractures or other rock mass discontinuities.

It is assumed that the regional groundwater flow divides coincide with surface water divides, i.e. groundwater flows from areas of higher to lower elevation, which is supported by hydraulic head data collected at monitoring wells located in the areas of each proposed open pit. Permafrost conditions (primarily located on north and west facing slopes) and low hydraulic conductivity overburden sediments in upland areas may reduce infiltration of surface water and therefore recharge to the bedrock aquifer below. All waters discharged from the site ultimately end up in the main surface water receiving body, the South Klondike River, south of the site.

The chemical composition of groundwater strongly depends on the local and upgradient aquifer lithologies. As groundwater flows through the aquifer, it assumes and continuously evolves a characteristic chemical composition due to interaction with the aquifer matrix. As such, a groundwater sample represents the local and upstream aquifer conditions, and its composition is a function of aquifer lithology, solution kinetics, water residence time, mixing, and groundwater flow patterns. All groundwater samples from the Property have a similar hydrogeochemical composition with slight differences between samples due to sample location. All groundwater samples are of a calcium and/or magnesium dominant cation type, and bicarbonate and/or sulphate anion type. Generally, all proposed open pit areas had at least one natural exceedance of Yukon Contaminated Sites Regulations (CSR) Aquatic Life Standards for routine parameters or dissolved metals.

Groundwater monitoring wells were installed hydraulically up gradient and down gradient at five of the proposed open pits and screen intervals completed at the proposed maximum depth of pits. Based on observed groundwater elevations, dewatering is expected to be required in all proposed open pit areas. However, dewatering volumes will depend on the location of the proposed open pits and is expected to vary considerably. Fosters would likely require the most significant dewatering with approximate inflow estimates based on simplifying analytical methods that suggest inflow rates to the Lower Fosters proposed open pit in the order of magnitude of up to several thousand cubic metres per day. Estimated inflow rates to the other open pits would likely be much smaller in the order of a few or up to hundreds of cubic metres per day. It is assumed that most in pit water will be pumped out to maintain operations and stability.

16.4 Haul Roads

The mining operation will require firstly re-establishing the existing haul road to Lower Foster, followed by re-establishing the existing haul roads to eastwards to Lucky. The work required involves removal of vegetation and growth media on the existing haul road, blading and grading, construction of an outside safety berm, ditching for proper road drainage, and surfacing the road with gravel. Culverts will be installed at the watercourse crossings and the road built up to provide a proper sag vertical curve profile. A minimum of 7 culverts are required along the haul road from Heap Leach to Lucky. It is anticipated that the existing haul roads will be wide enough for 2 lanes of traffic after completion except for a short single lane section between Golden and Lucky that is constrained by the existing steep topography.

The design for haul roads are based on limited information and will need to be reviewed at future studies including additional field investigation and site visits. Some of the parameters used in design of the haul roads are similar to previous values used for the existing mine such as cut and fill slope angles. The design parameters for the conceptual design of haul roads are listed below:

7. Travel road width of 13.7 m (2 times truck width plus space for safety berm);
8. Safety berm 1.4 m high (3/4 truck tire height);
9. Ditch depth of 1.5 m minimum;
10. Minimum design speed of 40 km/h;
11. Maximum profile grade of 11%;
12. Road embankment 1.4 m high plus 200 mm of surfacing gravel;
13. Cut slope at 1.0H : 1.0V and fill slope at 1.5H : 1.0V;
14. Road crowned at center and sloped at 3% cross fall; and
15. Minimum diameter of culverts is 800 mm.

In addition to re-establishing existing haul roads, three (3) new haul roads will be constructed, namely 1.3 km of haul road to East Big Rock, 2.0 km to West Big Rock and 0.7 km to Bohemian and Schooner. The haul road to East Big Rock follows the rolling terrain in a northwesterly direction from the main haul road at Pacific. There is a probability that permafrost will be encountered as the area is south facing but it is expected that the permafrost will be discontinuous. The conceptual design have incorporated fill embankment through the areas where permafrost is likely to occur. The haul road to West Big Rock starts from the main haul road at Heap Leach and follows the gentle slope down in a northwesterly to the valley. The cross section design is primarily embankment fill for the entire length of the haul road. It is not anticipated that the haul road will encounter permafrost. Fill source for the haul roads to East and West Big Rock is anticipated to come from run-of-mine waste from West Big Rock.

The haul road to Bohemian and Schooner traverses the relatively steep slope south of Lucky to the valley below, crossing over the watercourse leading to Lucky Creek, before ascending gradually to the Bohemian pit. A significant portion of the haul road is along the north face slope with a high probability of permafrost. For this reason, the conceptual design avoided any cuts which resulted in high fill volumes required for this haul road. Preliminary estimates indicate that there will be sufficient quantities of mine waste rock for the fill and haul road construction. It has been assumed that haul roads will be constructed from NAG waste rock made available from prestripping waste at Schooners. It is assumed that gravel surfacing material will be sourced from mine waste rock or heap leach material, screened and crushed by a mobile crushing unit.

It is anticipated that the existing mine site service roads will be rehabilitated for light vehicle usage. The estimate includes grading of approximately 3 km of road to a width of 6 m and surfacing with 75 mm depth of gravel. The 2 km existing access road to Laura Creek pump station will also be re-graded for light vehicle and machinery usage.

Haul road widths are designed to provide safe, efficient haulage, and to comply with the following BC mines regulations:

- For dual lane traffic, a travel width of not less than three times the width of the widest haulage vehicle used on the road, plus an allowance for shoulder barrier(s).
- Where single lane traffic exists, a travel width of not less than two times the width of the widest haulage vehicle used on the road, plus an allowance for shoulder barrier(s).
- Shoulder barriers of at least 3/4 of the height of the largest tire on any vehicle hauling on the road, placed along the edge of the haulage road wherever a drop-off greater than 3 m exists; the shoulder barriers are designed at 34° slope (slightly less than the angle of repose); the total road width equals the barrier width plus the travel width.

Based on the use of Komatsu HD605s or Hitachi EH1100, as the widest haul truck to be used, the haul road design basis is as follows:

- 4.75 m truck width used (may vary depending on truck selected)
- double lane highwall haul road allowance: 18.5 m
- single lane highwall haul road allowance: 13.7 m

The following Figures illustrate the single and double highwall haul road cross sections used on pit and waste dump designs.

Figure 16–1: Design of Single Lane Haul Roads

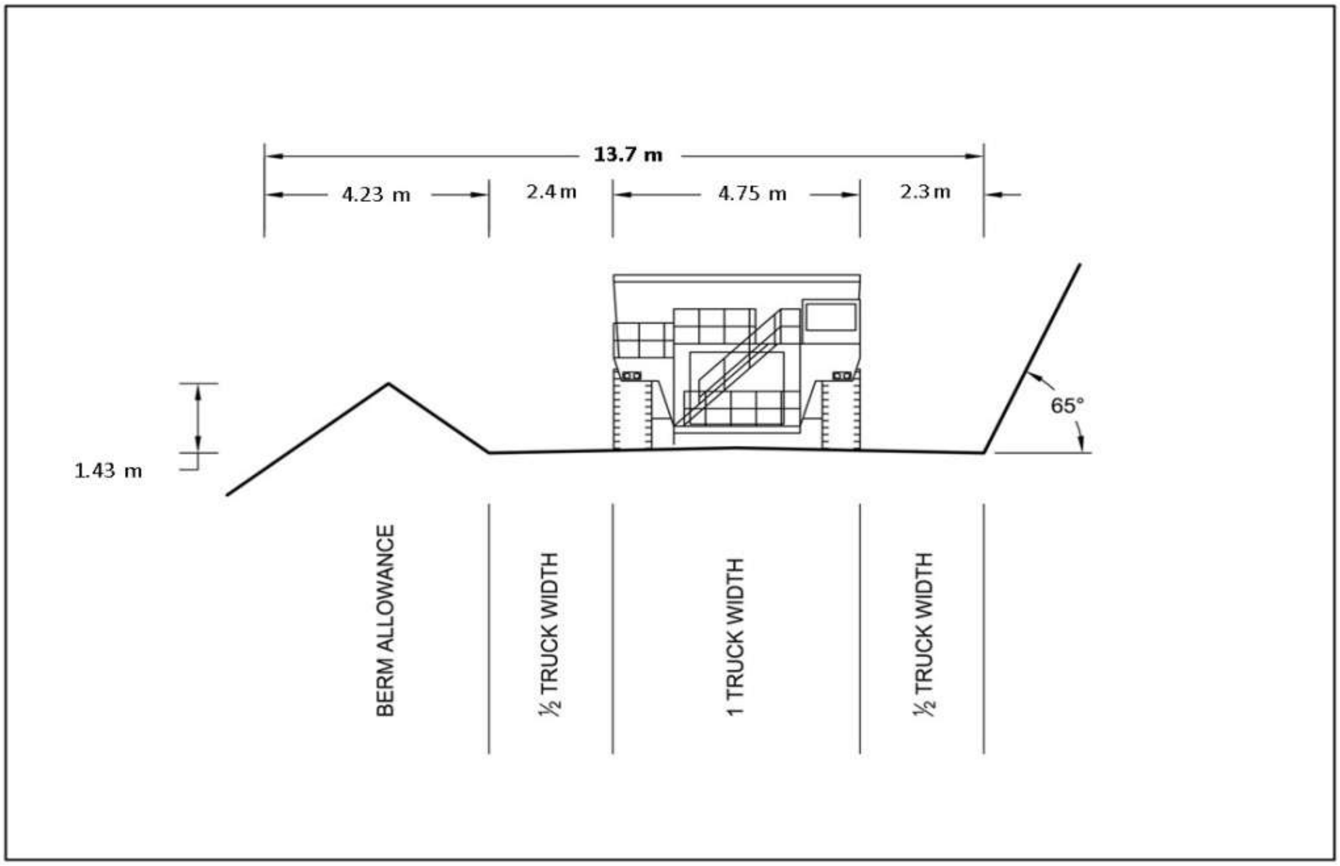
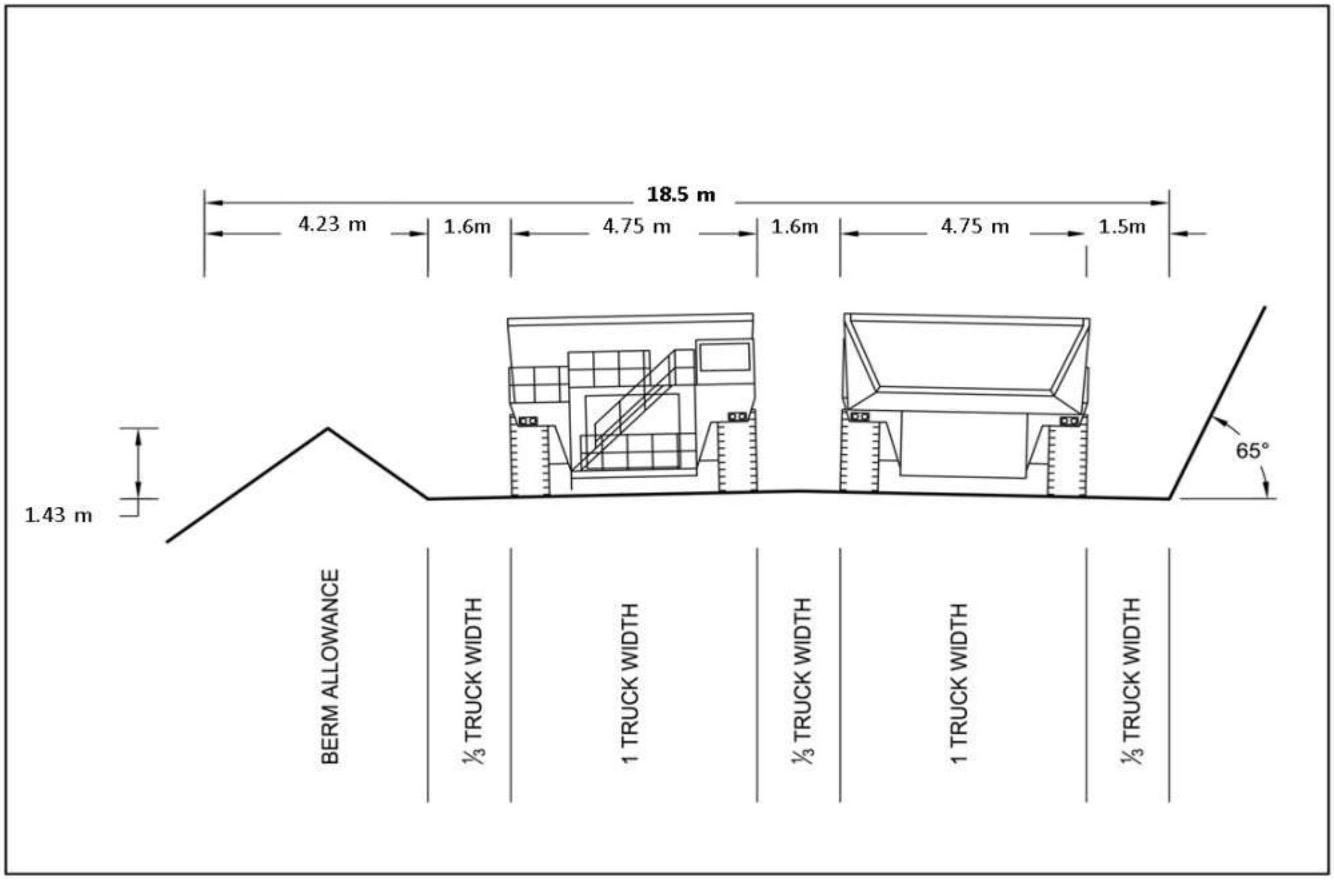


Figure 16–2: Design of Dual Lane Haul Roads



16.5 Design of the Open Pits

The design of the open pits was done in Gems software. The open pit design involved the creation of benched slopes based on the Whittle results and the design of haul roads within the pits, to make mining possible. Single lane haul roads of 13.7 m wide as shown in Figure 16–1 were used in the open pits, with double lane for external haul roads. A minimum mining width 15 m has been used for final pit layouts.

The parameters used in the open pit design are detailed below:

- Flinch height: 6 m
- Bench height: max 18 m (3 benches)
- Bench face: maximum 65° from horizontal
- Bench style: triple benching
- Berm/catch bench width: 8 m

The following pit-specific slope design guidelines were used, where applicable:

Kokanee, Golden, Lucky, Bohemian, West and East Big Rock

All slopes must be a maximum of 48° from the horizontal.

Lower Fosters

South side of the pit must have slopes that are a maximum of 48° from the horizontal. The main access ramp has been placed on northern side of the pit to ensure that the slope is lowered, though the ultimate pit slope angle is subject to the height of the slope such that the slope configurations are as shown in Table 16-6.

Table 16-6: Slope Design Parameters for Fosters Pit

Slope Height	Maximum Slope Angle
<80	48°
90	46.5°
100	45.5°
110	43.5°
120	40.2°
140	37.0°

Schooners

All slopes must be a maximum of 48° from the horizontal. Though, the far western side of the pit should have a slope not exceeding 45°.

16.5.1 Pit Designs Results

Figure 16–3 to Figure 16–12 show the pit designs for the 8 deposits from West to East.

Figure 16–3: West Big Rock Ultimate Pit Design

Figure 16–4: East Big Rock Ultimate Pit Design

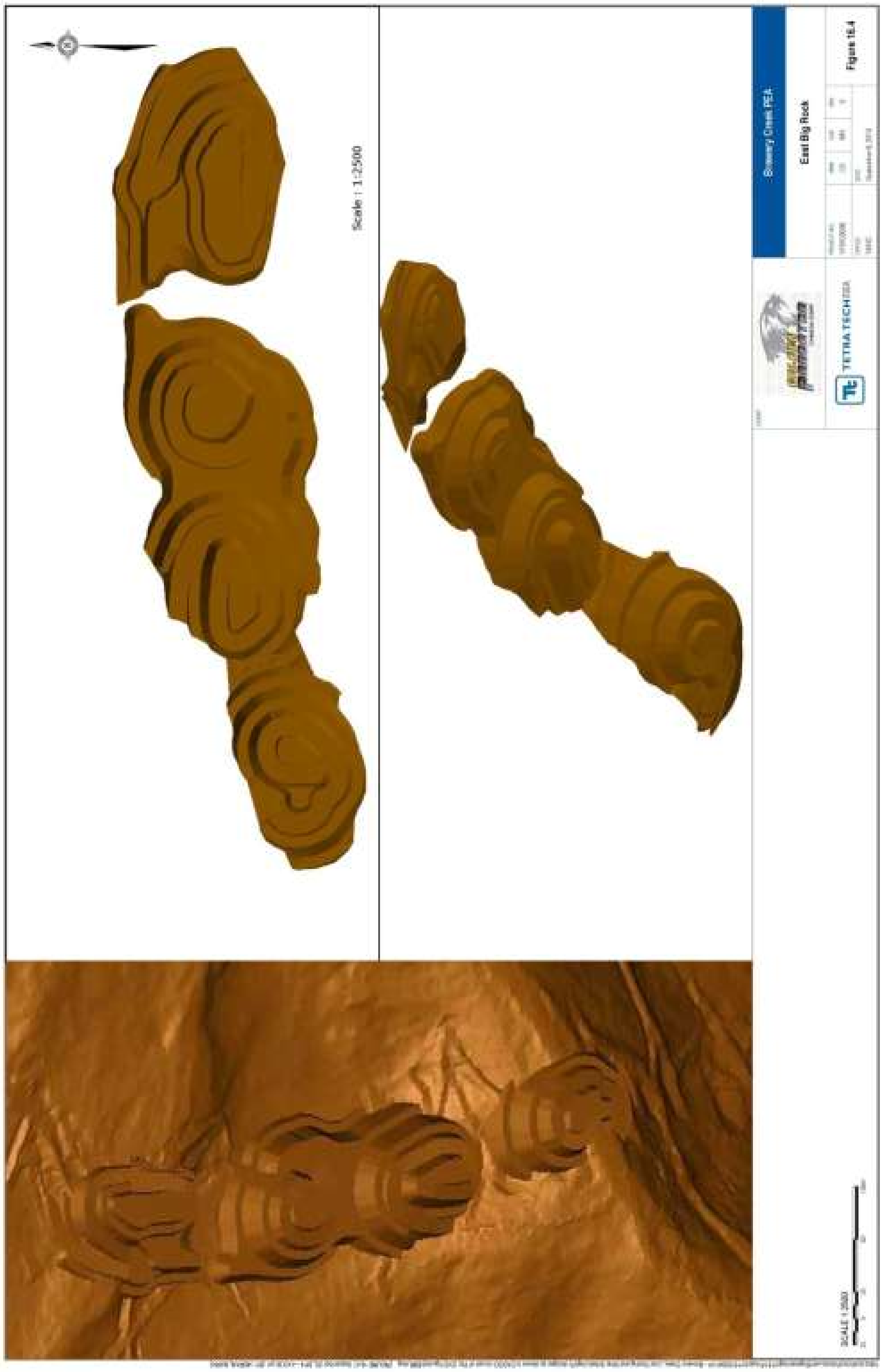


Figure 16–5: Fosters Ultimate Pit Design

Figure 16–6: Kokanee Ultimate Pit Design

Figure 16–7: Golden Ultimate Pit Design

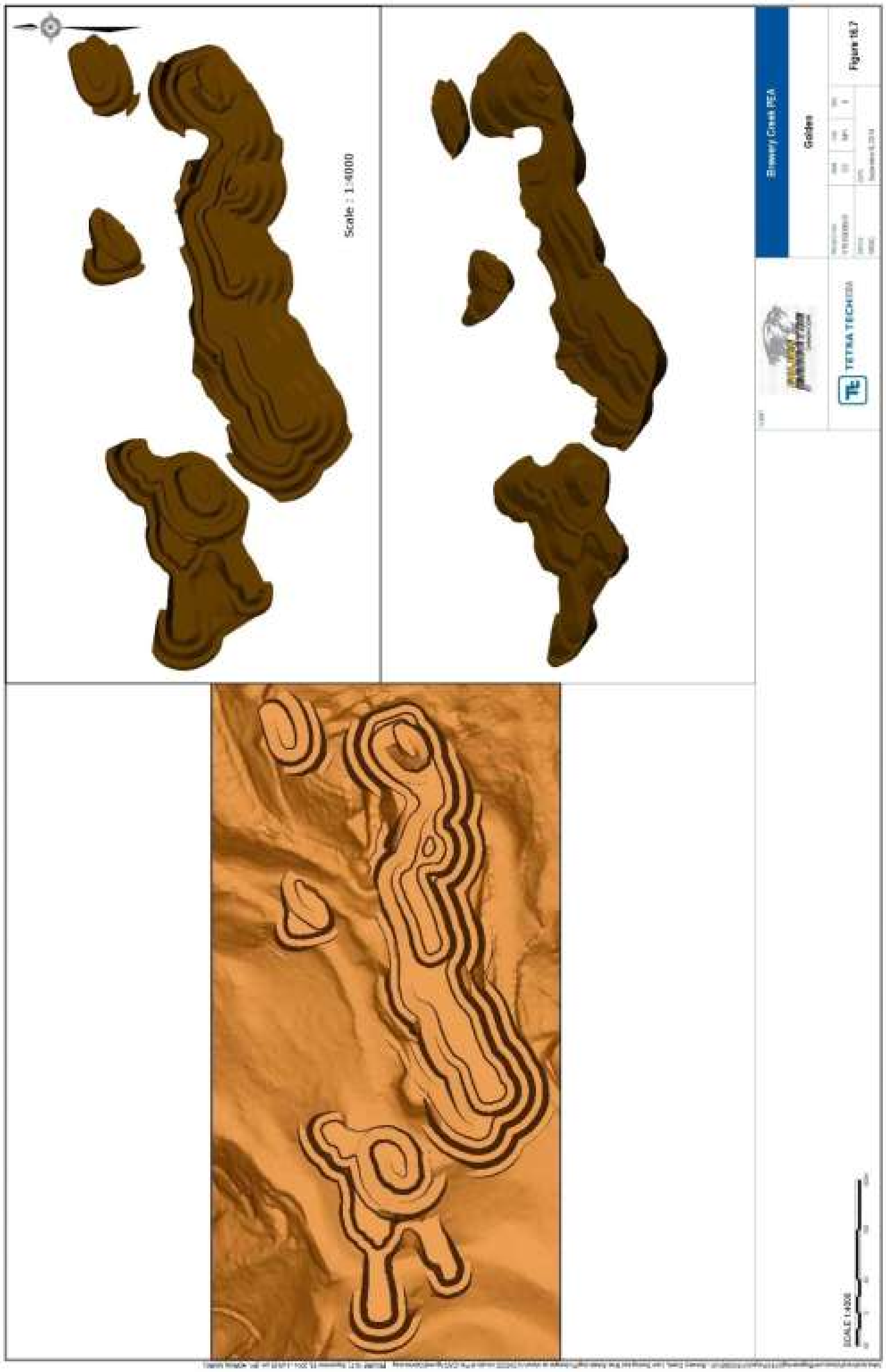


Figure 16–8: Lucky Ultimate Pit Design

Figure 16–9: Bohemian Ultimate Pit Design

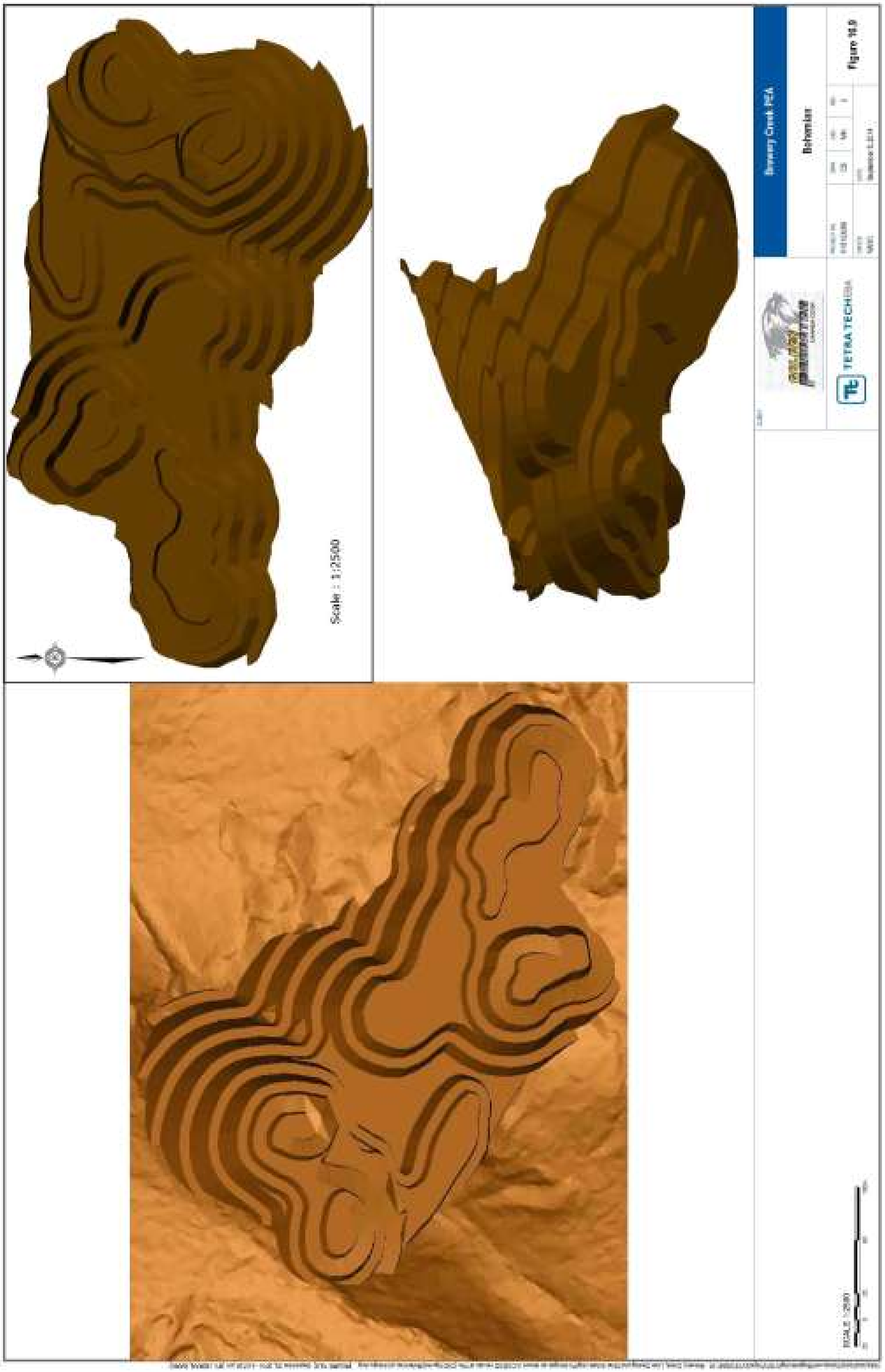


Figure 16–10: Schooners Ultimate Pit Design

16.5.2 Mine Plan and Mine Schedule

A strategic schedule has been generated for the mining of 8 deposits included in the PEA. The schedule was generated using the open pit designs for each deposit, and a Whittle module for multi-mine scheduling. The module determines a strategy to mine the deposits based on trading-off grades and mining costs. The mining and processing parameters used to generate the schedule were the same as those used to generate the Whittle pit shells as described in section 16.2. Additional scheduling parameters were applied to generate a monthly production schedule over the life of mine. These are:

- Annual processing feed rate - 1,725,000 tonnes
- Monthly processing feed rate - 215,625 tonnes
- Annual total mining limit - 6,600,000 tonnes
- Monthly total mining limit - 825,000 tonnes
- Days processing per year - 230 days
- Days mining per year - 230 days
- Months processing per year - 8 months
- Pre-strip period - 1 year
- Discount rate used as annual - 8%
- Discount rate used as monthly - 0.64%
- Dilution - 5%
- Mining losses - 5%

16.5.3 Scheduling Results

Table 16-7 below shows the mining of process feed on an annual basis throughout the life of mine. Table 16-8 provides a breakdown of the gold estimated as process feed by pit mined in kg. Table 16-9 provides a breakdown of waste tonnes mined each year, including waste material from the old heap leach. The waste from the old heap leach would be material below the cut-off grade of 0.3 g/t. If this material is geotechnically suitable it may be used for haul road construction, heap leach liners or fill material for mine infrastructure pads.

Table 16-7: Ktonnes (Kt) Processed from the Pits and Reprocessed from the existing Heap Leach Pad for Each Year of Operations

Pit Area	Total mine-life	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Schooner	1,044	1,044								
Fosters	1,275	582	452	241						
Bohemian	1,577		387	1,190						
Golden	878			305	573					

Table 16-7: Ktonnes (Kt) Processed from the Pits and Reprocessed from the existing Heap Leach Pad for Each Year of Operations

Pit Area	Total mine-life	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Kokanee	1,243				887	356				
WBR	809					809				
EBR	465						408		57	
Lucky	2,973						317	1,526	1,130	
Total process feed from Pits kt	10,264	1,626	840	1,736	1,460	1,165	726	1,526	1,187	
Total processed from old heap leach kt [1]	**4,180**	**102**	**886**	**71**	**267**	**561**	**999**	**200**	**108**	985
Total processed kt	**14,445**	**1,728**	**1,726**	**1,807**	**1,727**	**1,726**	**1,725**	**1,726**	**1,295**	985

[1] The schedule includes material from the existing heap leach, for which an estimate for the tonnes is available, though grade data is based on limited information. Tetra Tech has not scheduled the reprocessing of the existing heap leach but has considered that where the scheduled tonnes from the open pit mining are less than the processing capacity for that period, the available process capacity will be filled through reprocessing the heap leach material. A 0.3gpt cut-off has been applied to the re-processed heap leach material.

Table 16-8: Gold mined in kg from Pits and Old Heap Leach Over Life of Mine

Pit Area	Total mine-life (kg)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Schooner	2,157	2,157								
Fosters	2,067	1,003	725	339						
Bohemian	1,919		376	1,543						
Golden	1,176			352	824					
Kokanee	1,321				946	375				
WBR	945					945				
EBR	496						447		48	
Lucky	3,764						320	1,856	1,588	
Total gold in process feed from Pits kg	13,845	3160	1,101	2,235	1,770	1,320	767	1,856	1,636	
Total processed from old heap leach kg	3,219	79	682	55	206	432	770	154	83	759
Grade of process feed from pits	1.35	1.94	1.31	1.29	1.21	1.13	1.06	1.22	1.38	
Overall process feed grade	1.18	1.87	1.03	1.27	1.14	1.02	0.89	1.16	1.33	0.77

Table 16-9: Brewery Creek Annual Waste Mining Schedule in Kt

Pit Area	Total mine-life (kt)	Pre-stripping	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Schooner	8,198	6,116	2,082							
Fosters	5,599	424	1,688	2,644	843					
Bohemian	4,960			3,118	1,842					
Golden	2,776				1,790	985				
Kokanee	3,908					3,581	327			
WBR	4,167						4,167			
Lucky	11,677						100	3,897	5,051	2,629
EBR	2,235							1,980		255
Total waste mined from pits kt	43,520	6,539	3,771	5,762	4,476	4,567	4,593	5,876	5,051	2,884
Total waste from old heap leach kt	**3,366**	582	713	752	708	611				

16.6 Waste Rock Storage

Ten waste rock storage locations have been planned and designed for the PEA. The waste rock schedule was developed as a strategy for the disposal of waste into the allocated conceptual waste dump facilities and potential backfill of mined out pits.

Monthly waste rock tonnes were produced in the scheduling process using Whittle software. In line with various densities for waste materials, the volumes of the material where estimated for in situ as Bank Cubic Metres (BCM) and as placed in waste rock dumps as Loose Cubic Metres (LCM) after a swell factor was applied.

In reference to work completed by Access Consulting Group in a January 2013 submission titled “Brewery Creek Mine Reactivation Project Moosehead, Fosters and Big Rocks Deposits,” (GPCC, 2013) the waste rock scheduling process considered waste rock lithology in accordance with the rock classification contained in the report: intrusive rock types were considered low risk of long term acid generation with sediment rock types considered to have low to moderate risk of long term acid generation. Any unknown rock types were treated similarly to the sediment characterization.

More work needs to be undertaken to determine the acid generating risk of the various rock types and the subsequent risk of the various dumps with rock type combinations.

16.7 Geometry of Potential External Waste Storage Embankments

For the purposes of the PEA, two primary embankment types are being proposed for external waste disposal on the project: headwater valley fill and sidehill fill. Table 16-10 shows the input parameters for physical waste rock criteria. The embankments would be constructed by end dumping methods, ascending construction or descending construction methods. Table 16-11 lists the assumptions used as basic criteria for estimation of typical embankment and footprint dimensions.

Table 16-10: Input Assumptions for Physical Characterization of Rock Type

Parameter	Oxide-Sediment	Oxide-Intrusive (LAQM)	Un-oxidized Sediment	Un-oxidized Intrusive (AQM)
Rock types	Argillite, sandstone/greywacke, siltstone	Limonite altered quartz-monzonite	Argillite, sandstone/greywacke, siltstone	Quartz monzonite
Estimated Bulk Density of Compacted Waste Rock (t/m³)	2.06	2.06	2.14	2.11
Swell Factor	0.8	0.8	0.8	0.8
Angle of Repose	34°	34°	34°	34°

Table 16-11: Input Assumptions for Embankment and Footprint Dimensions

Parameter	Headwater Valley Fill	Sidehill Fill
Foundation Stability Rating requirement	I-III	I-III
Minimum berm width (m)	12	12
Maximum lift height (m)	15	15
Maximum overall Slope Angle	26.6°	26.6°
Construction method	End dump/descending benching	End dump/ascending benching

The allocation of waste rock from each pit and to each conceptual waste rock dump is presented in Table 16-12 below.

Table 16-12: Allocation of Waste Material from each Open Pit to each Dump Location in LCM ($x10^3$)

Type of Waste Dump		Construction	External dumps	External dumps	External dumps	External dumps	External dumps	Back fill	Back fill	Back fill	Back fill	Back fill
			CWD-01	CWD-03	CWD-04	CWD-06	CWD-07	Schooner	WBR	Lower Fosters	Golden	Kokanee
Total waste mined (LCM)	21,115	1,254	2,003	2,738	3,826	2,406	656	2,372	1,063	1,660	1,209	1,925
Schooner	3,917	1,064			430	2,406	16					
Fosters	2,656	190		2,466								
Bohemian	2,372							2,372				
Golden	1,339				821		518					
Kokanee	1,933			273						1,660		
WBR	2,003		2,003									
Lucky	5,831				2,575		122				1,209	1,925
EBR	1,063								1,063			

The rock types and the resultant risk relating to acid production (GPCC, 2013) from the waste have been used to classify the rock as shown in Table 16-13.

Table 16-13: Waste Rock Types Placed in each Conceptual Waste Rock Dump or Open Pit Backfill

Area	Waste dump used	LCM (x103) Placed	Low risk material (intrusive)	Low to moderate risk material/ Sediment/ Uncertain	Total rock to be placed	Designed Capacity	Designed Capacity remaining (deficient)
Construction	Construction	1,254	1,254	0	1,254	1,018	(236) [1]
External dumps	CWD-01	2,003	1,285	718	2,003	2,007	4
	CWD-02					1,154	1,154
	CWD-03	2,738	466	2,272	2,738	2,789	51
	CWD-04	3,826	3,331		3,826	3,827	1
	CWD-05	0				677	677
	CWD-06	2,406		2,406	2,406	2,408	2
	CWD-07	656		656	656	677	21
Back fill	Schooner	2,372	1,458	915	2,372	2,387	15
	WBR	1,063	333	730	1,063	1,068	5
	Bohemian	0				1,898	1,898
	Lower Fosters	1,660	941	719	1,660	1,668	7
	Golden	1,209		1,209	1,209	1,239	30
	Kokanee	1,925		1,925	1,925	1,936	11
Total		21,114	9,067	11,552	21,114	24,754	3,640

1 Construction capacity for waste is based on the construction material needed for haul roads. Tetra Tech EBA expects that additional capacity for use of waste in construction will occur throughout the project, including the crushing and agglomeration area, run-of-mine stockpile pad and other pads at the ADR, site offices and camp. Once further engineering has been done these figures can be updated to reflect the design capacity.

16.7.1 Waste rock storage facility designs

Figure 16–11 to Figure 16–15 show External waste rock storage facility designs from West to East.

Figure 16–16 to Figure 16–20 show waste storage facilities as backfill of mined out pits are shown from West to East.

Figure 16–11: Waste Rock Storage Facility CWD – 01

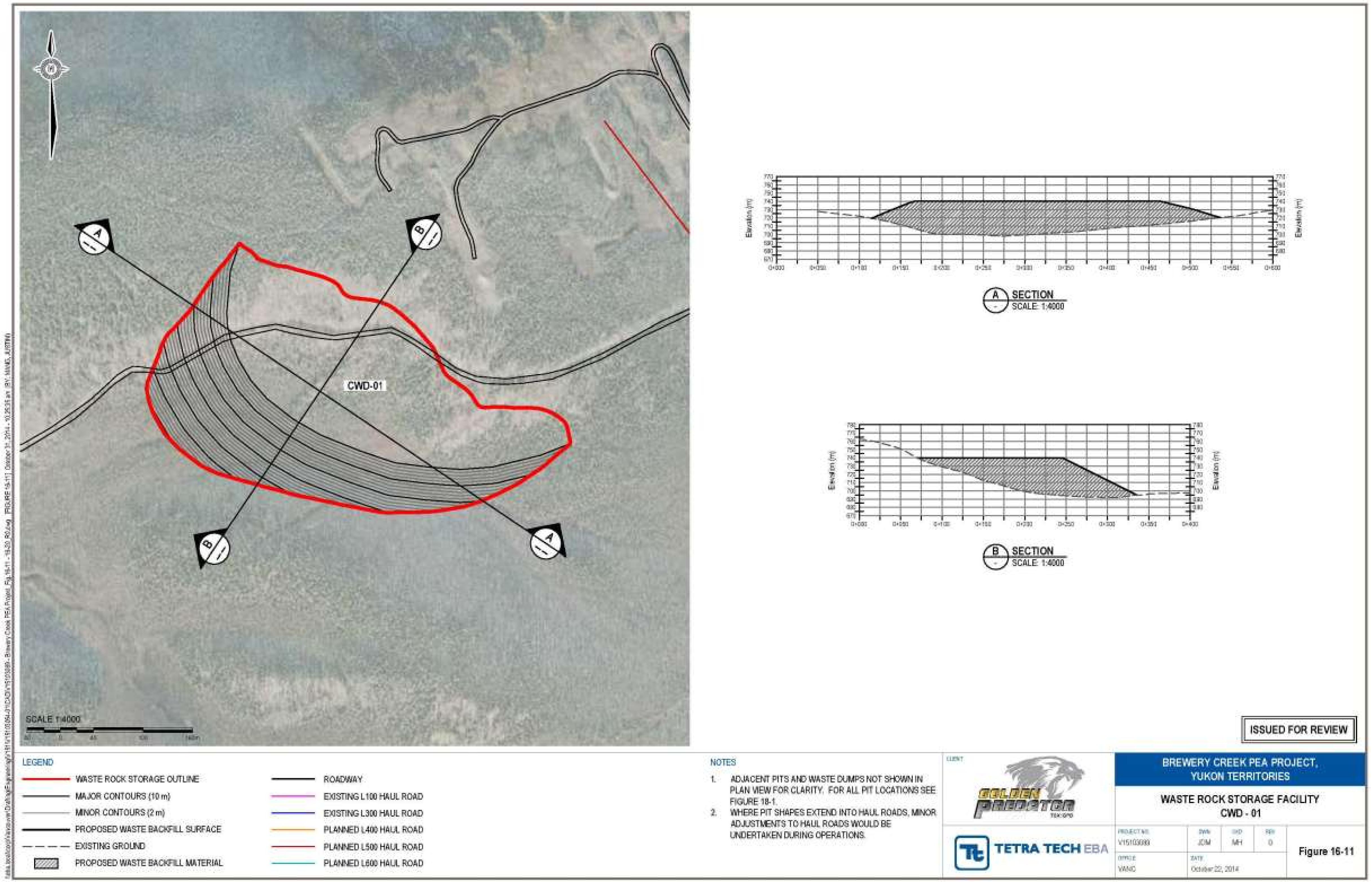


Figure 16–12: Waste rock storage facility CWD – 03

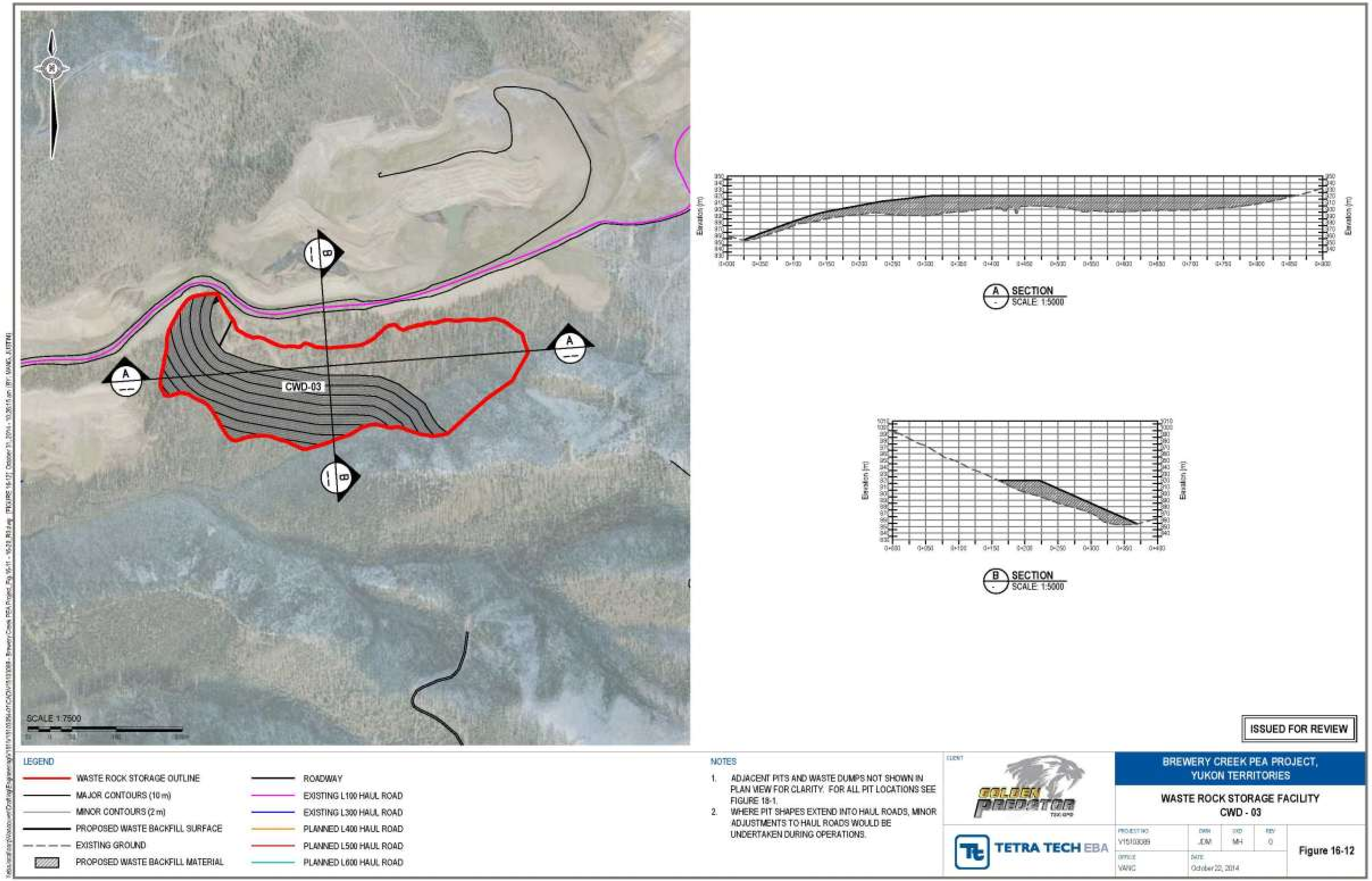


Figure 16–13: Waste rock storage facility CWD – 04

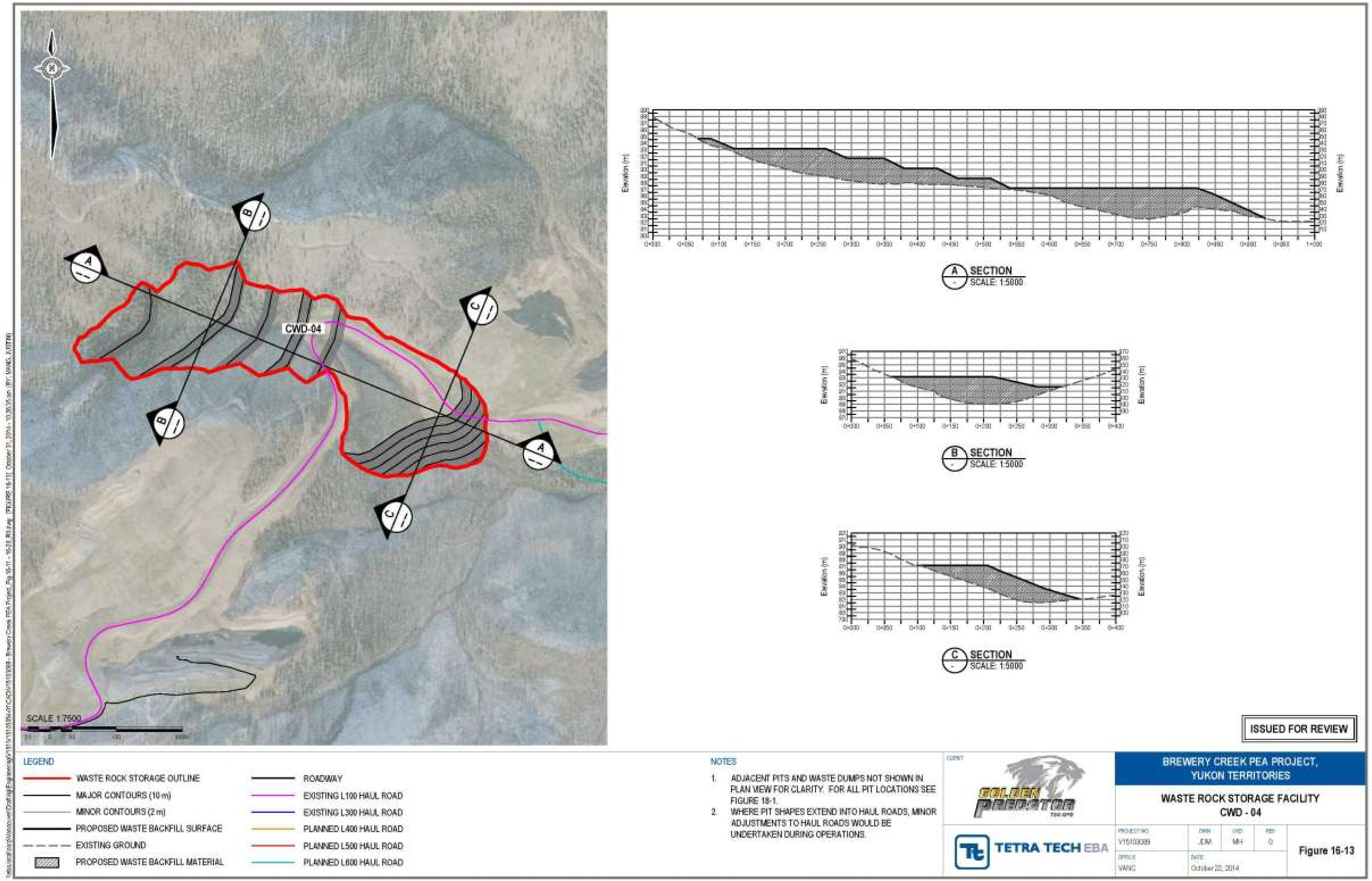


Figure 16–14: Waste rock storage facility CWD – 06

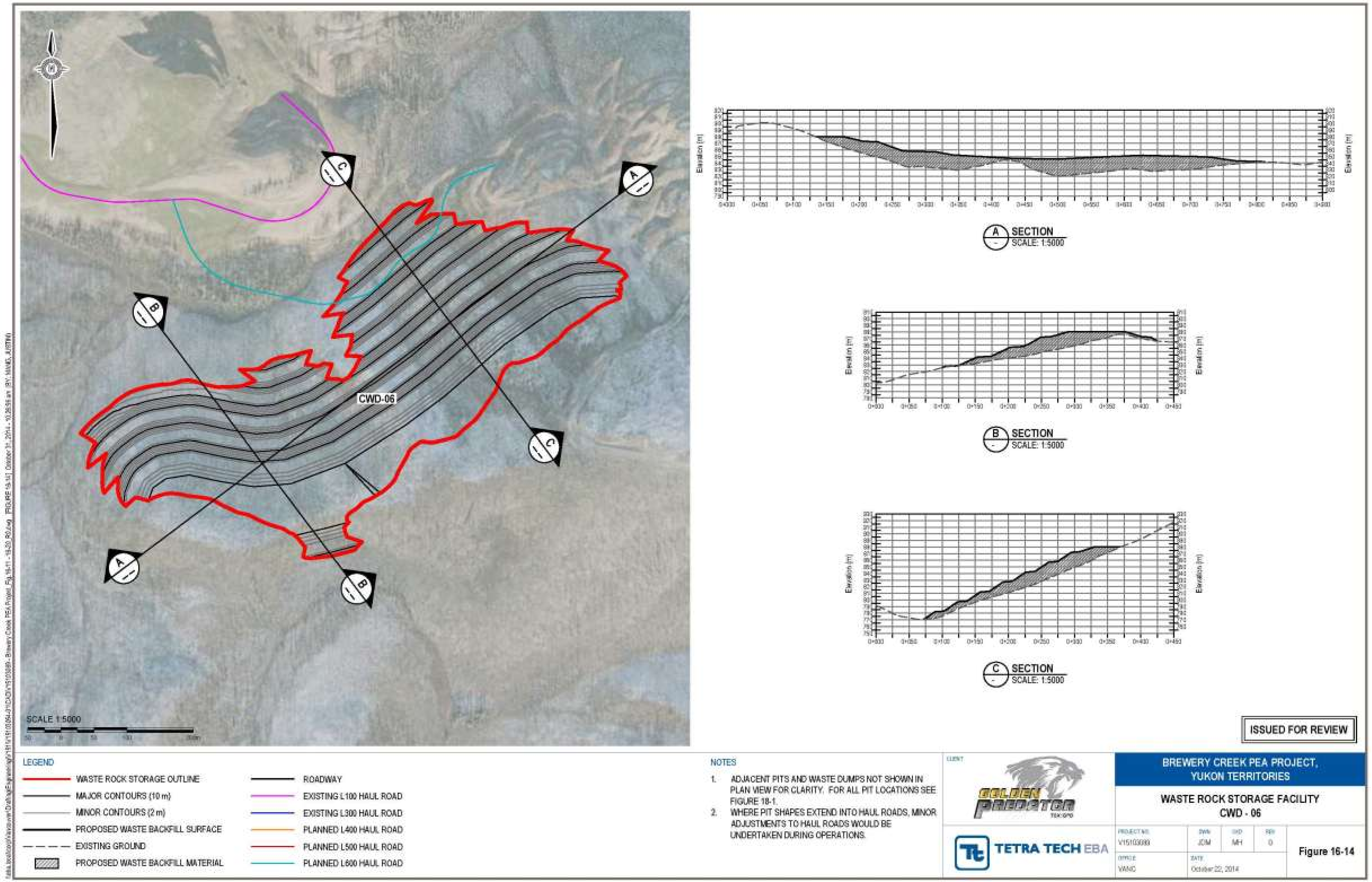


Figure 16–15: Waste rock storage facility CWD – 07

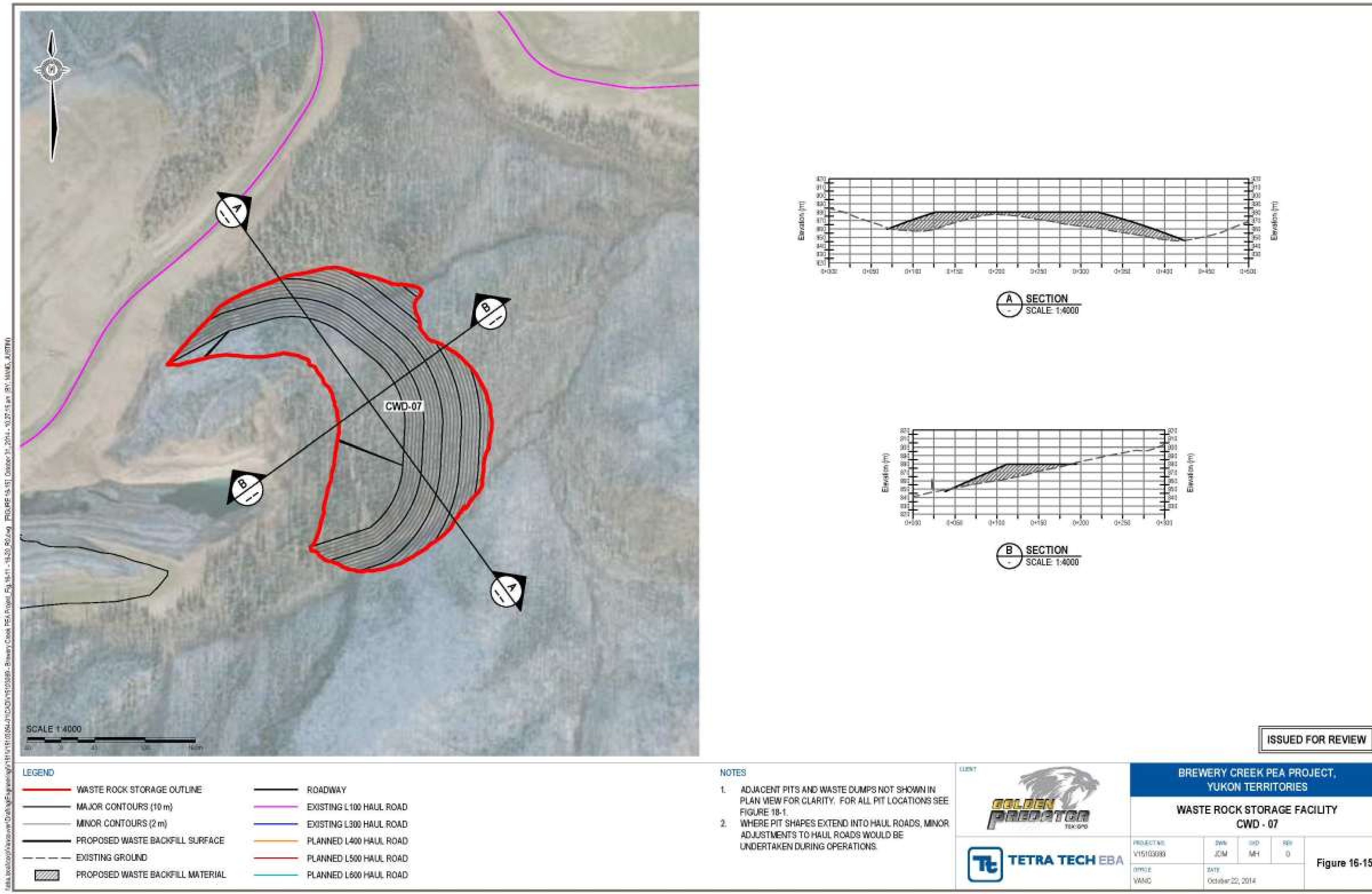


Figure 16–16: West Big Rock Backfill

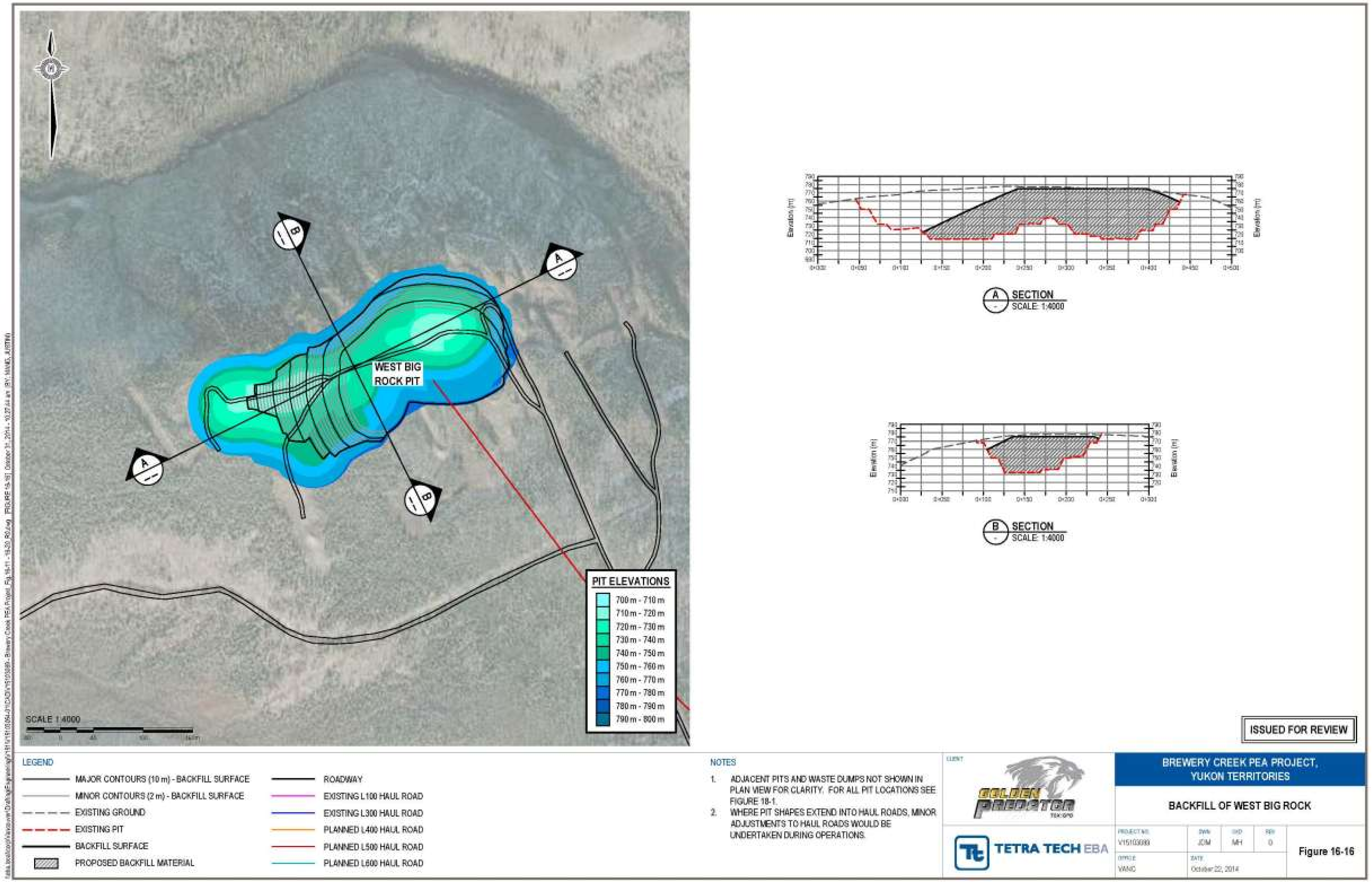


Figure 16–17: Lower Fosters Backfill

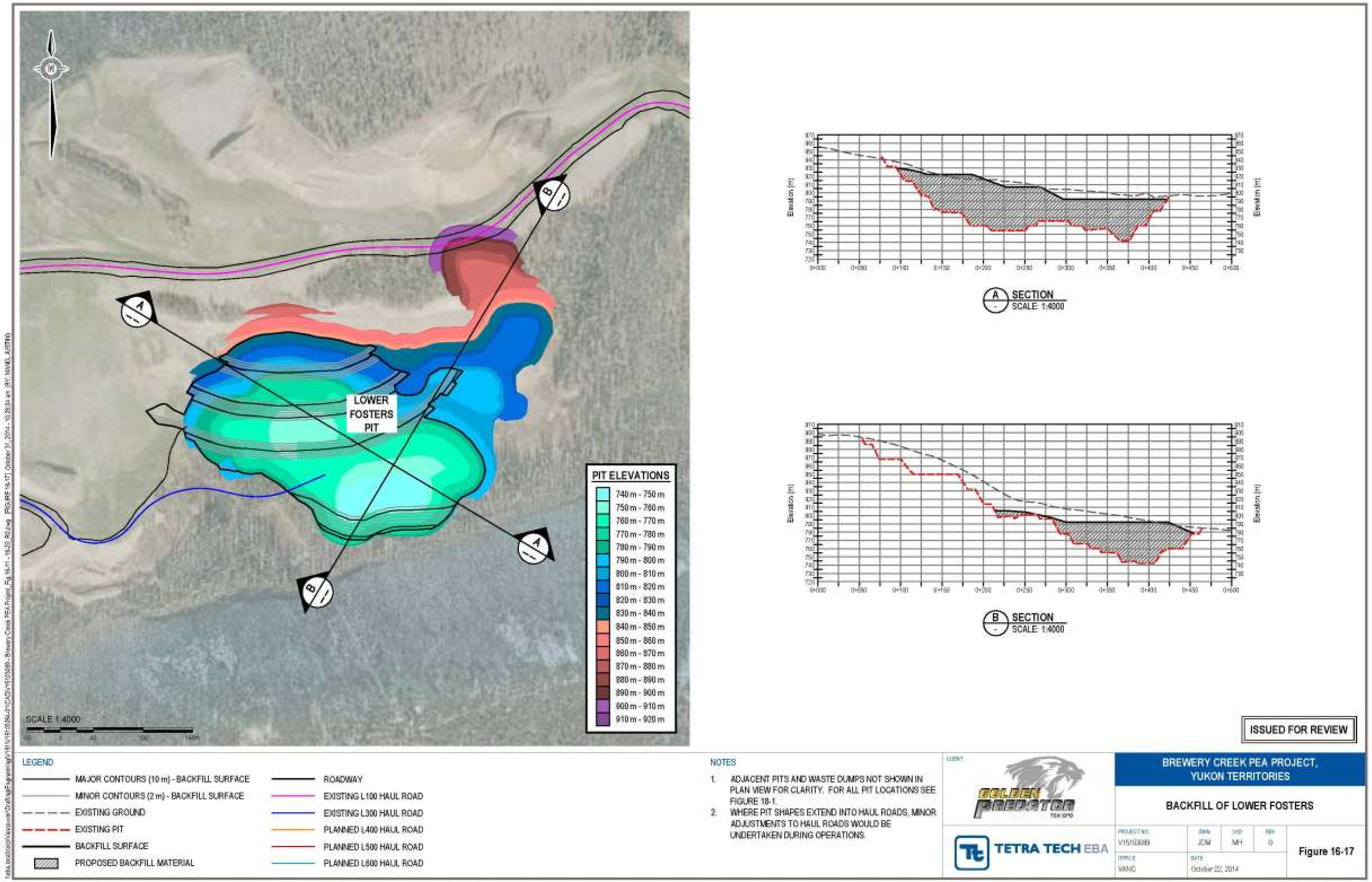


Figure 16–18: Kokanee Backfill

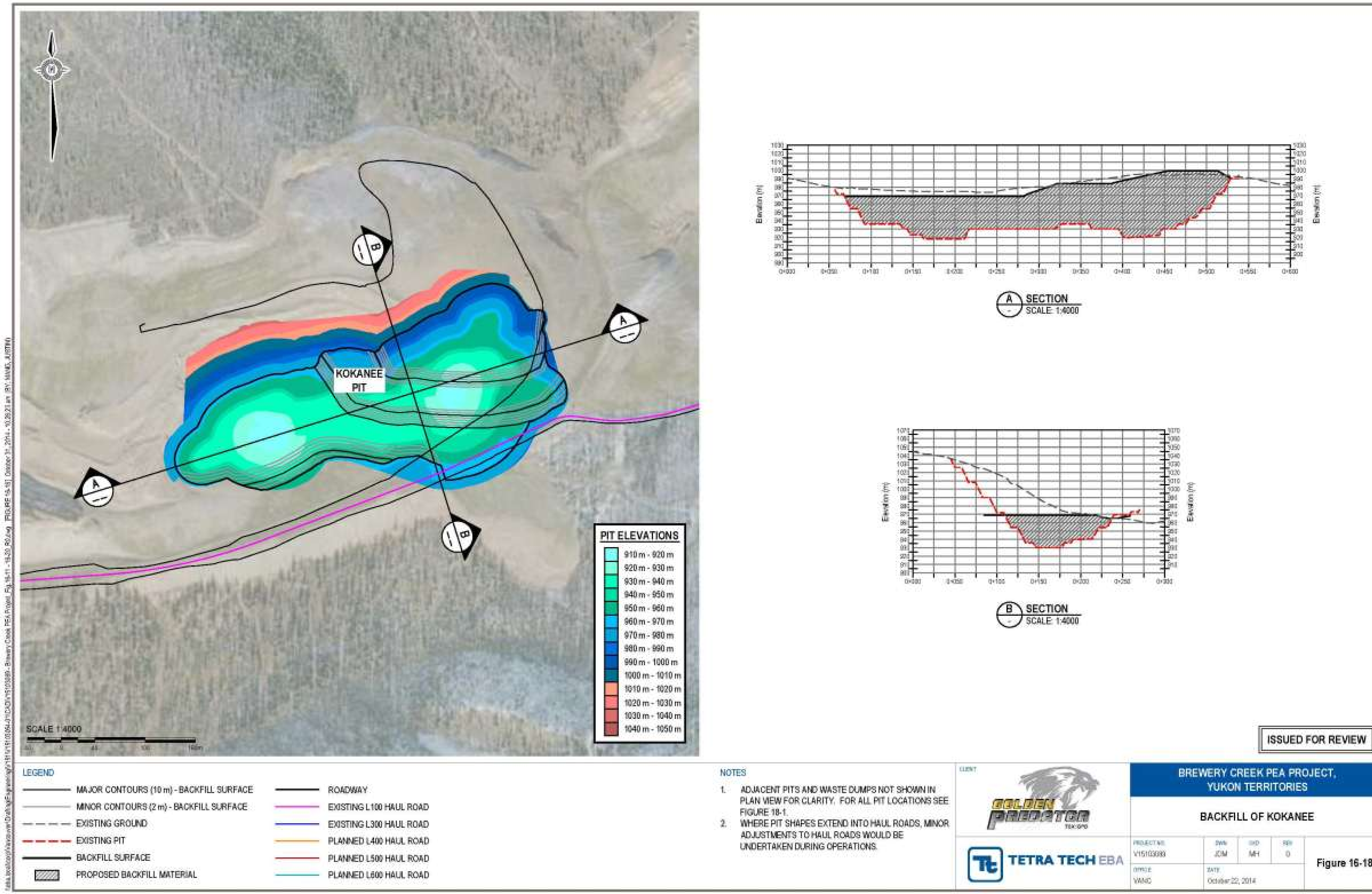


Figure 16–19: Golden Backfill

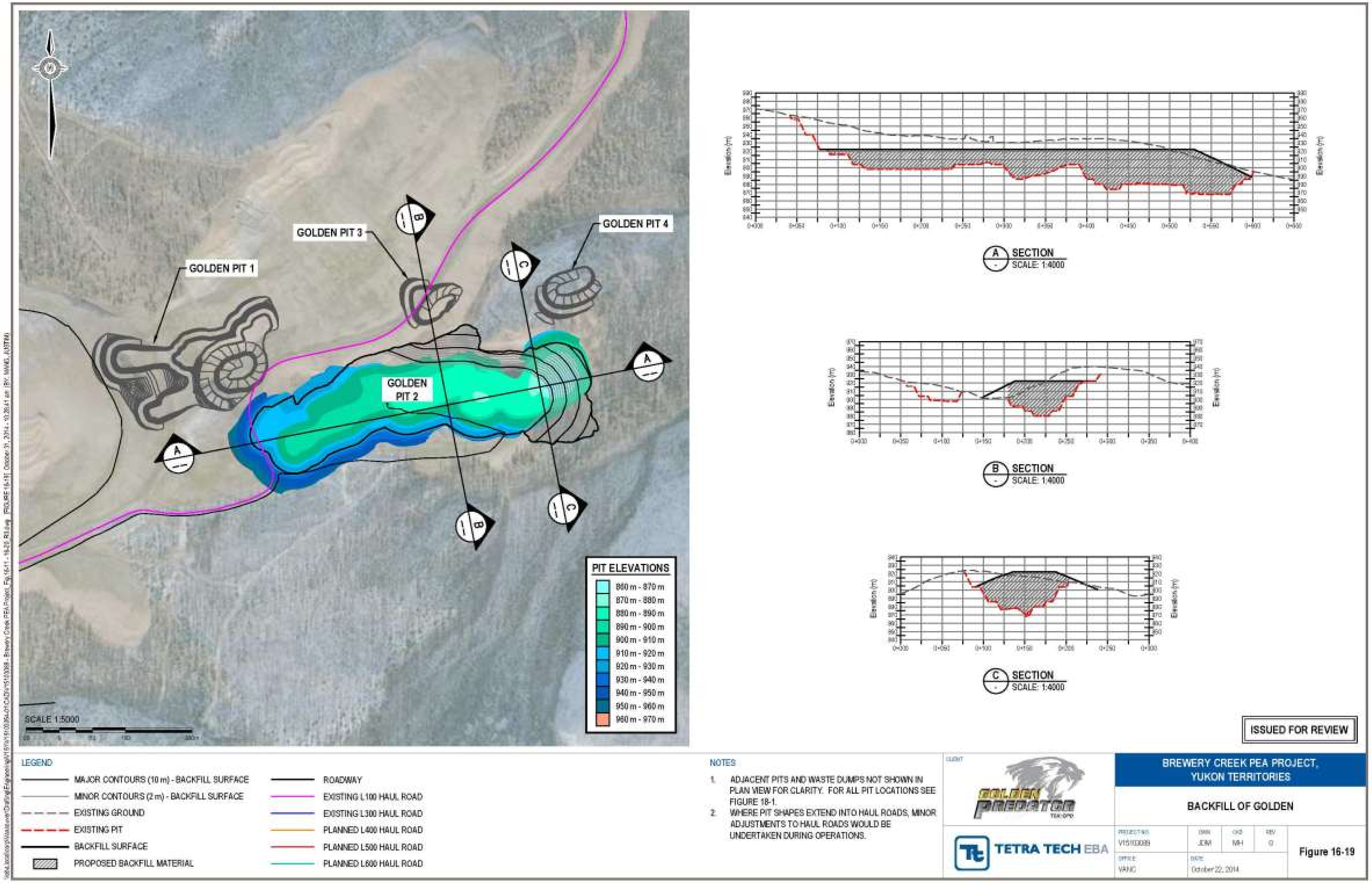
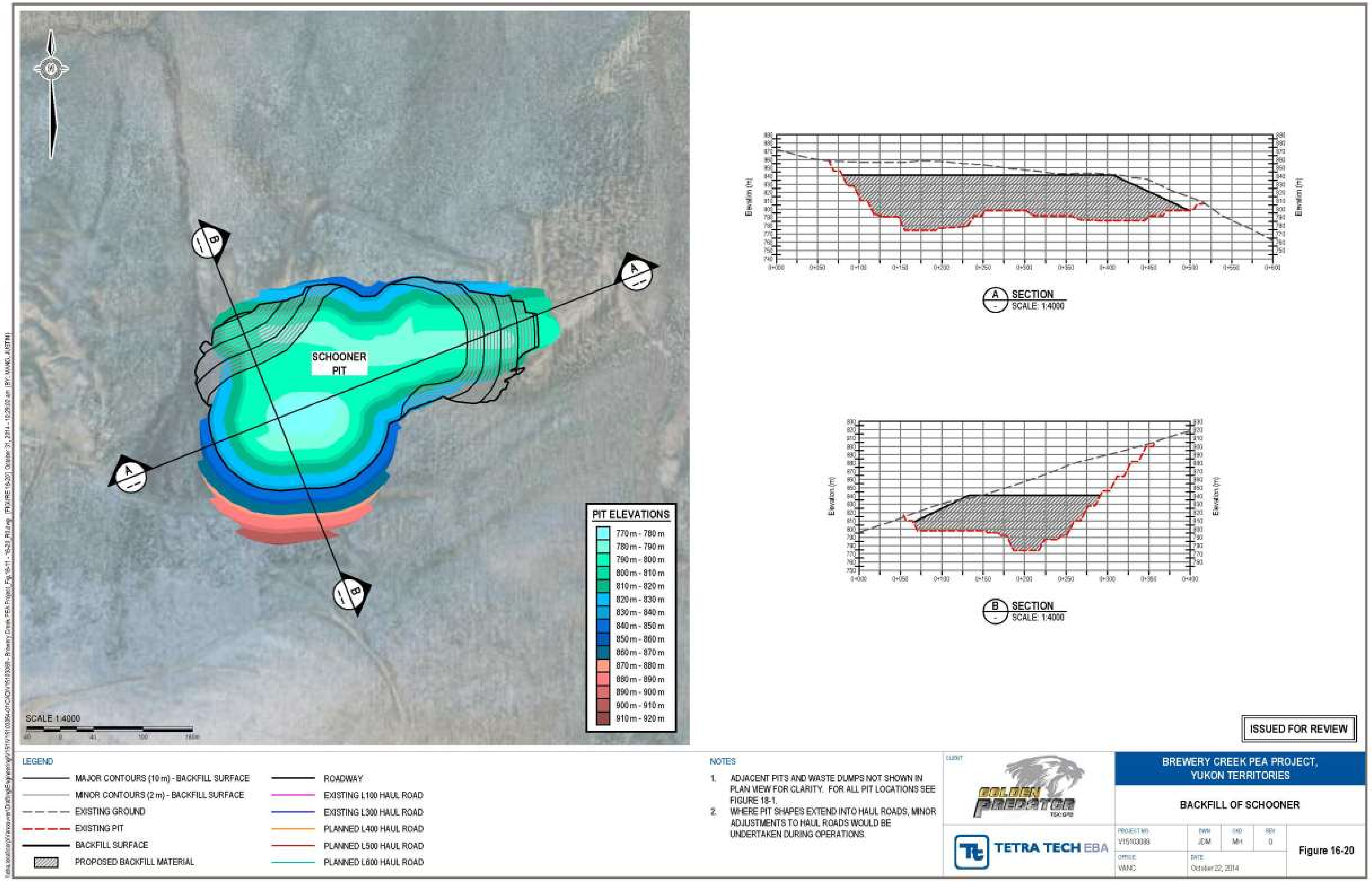


Figure 16–20: Schooner Backfill



16.8 Waste Rock Management

A desktop study was completed by Tetra Tech EBA in conjunction with a site investigation to determine the best locations for waste rock storage facilities within the Brewery Creek Property. Waste rock storage facility locations selected included previous mining areas and suitable areas in the vicinity of future mining areas.

Each waste rock storage facility location was evaluated based on a preliminary rating system that outlined the options for disposal of the waste rock generated from each of the proposed open pits. The strategy behind the rating system was to address the geotechnical, terrain stability, logistical and to perform a preliminary consequence analysis when determining the overall desirability of the waste rock storage facility location. Waste rock materials found to be reactive (i.e. potentially acid generating and metal leaching) will be dealt with on a case by case basis, ensuring environmental protection. Preliminary work on geochemical characterization of the waste rock was conducted by Access (2012) and is summarized in Section 20.2.1.1. Detailed drainage plans and hydrologic assessment will be completed for the construction of each respective dump.

Waste rock is planned to be used for construction purposes, depending on any acid rock and metal leaching potential of the material. New sections of road that cross a stream or creek will require rock fill to be placed around a culvert for continued drainage. Crossing fill volume will be limited by culvert length unless an in-stream coarse rock sub-grade fill is considered acceptable by Yukon regulatory agencies.

Two areas have been identified as suitable for the creation of more favourable haul road alignments by replacing the tight corners of valley crossings with straighter sections. These areas are the tight corner to the west of the Fosters pit and along the haul road alignment west of the Lucky pit.

16.8.1 Construction of Waste Rock Storage Facilities and Pre-development

In preparation for waste rock being placed on undisturbed ground, organic soils will be stripped and stockpiled for site reclamation purposes unless the site is underlain by permafrost. When the waste rock deposition site is found to be underlain by permafrost, trees will be removed and the organic mat of ground vegetation and underlying organic soils will be left undisturbed.

The waste dumps constructed on undeveloped ridge crests and slopes and small headwater valley fills will have a final side slope of 2 H:1 V. This assumption was based on historic design information available from when the property was owned by Loki Gold Corporation. The historic waste dump design included the excavation of organic matter to expose a firm foundation prior to construction. Historic waste dumps are in locations where permafrost is absent.

16.9 Mining Equipment Fleet

For the purpose of the PEA, a mining fleet has been selected based on production requirements and estimated equipment productivities. A list of the mining equipment selected and used for capital and operating cost estimates is included in Table 16-14.

Table 16-14: Mining Mobile Machinery Selected

Machine	Usage	Type	No.	Capacity	Rated kW
Blast hole drill	Drilling of 115mm holes for ore and waste rock	Crawler - DTH	2	45 m/hr	287
Shovel	Loading of process feed from pits	Crawler - Front/backhoe	1	6.3 m³	578

Table 16-14: Mining Mobile Machinery Selected

Machine	Usage	Type	No.	Capacity	Rated kW
Front end loader	Loading of waste	Wheel loader	1	8.4 m³	475
Front end loader	Pit services, rehandle and loading of old heap leach	Wheel loader	1	6.4 m³	
Dump truck	Hauling of process feed from pits to stockpile	Long distance hauler	3	80t	587
Dump truck	Hauling of waste from pits to waste rock dumps	Rigid frame dumper	5	40 m³ / 65t	551
Dump truck	Mining old leach pad and placing new material on leach pad	Rigid frame or articulated dumper	2	40 t	399
Bulldozer	Road construction, waste dump formation, clearing and grubbing and reclamation	Tracked	2	13.7	335
Water Truck	Dust allaying on haul roads	Rigid frame	1	10000 gl.	342
Grader	Haul road grading	Wheeled	1	4.3 m blade	165
Fuel and lube truck	Transport fuel and lubricants to mobile fleet	Truck mounted	1	2460 ltrs	82
Medium truck	Assist with maintenance of mobile fleet	Flatbed truck	1		62
Light vehicles	General supervision and management	Single cab	5		250
Pump	Dewatering of pits	Self-priming centrifugal diesel	1		90
Lighting sets	Lighting of mining area for night shift	Self-contained trailer units	2	10 kW	10
Low boy truck	Transport of drill, dozers and shovel around site	Horse and trailer	1		500
Tire handler	Handling truck tires for repairs		1		62
Compactor	Haul road construction	Sheep's foot	1		175
Welding truck	Assist with mobile machinery repairs	Truck mounted	1		62
Explosive truck	Deliver explosives to site	Truck mounted	1		180
Mobile crusher	Crush waste rock for haul roads	Mobile jaw crusher	1		165
Backhoe	Load crusher and general assistance	Tracked backhoe	1	1.2 m³	121

For the selection and estimation of the productivity of haul trucks, the haul routes for process feed and waste rock were used as a basis for cycle time estimates.

16.9.1 Haul truck cycle times

The cycle times for haulage were based on estimates of haul truck speeds on the various haul route segment alignments. The travel speeds along each alignment accounted for horizontal and vertical curves. The effects on travel speeds are described below.

16.9.1.1 Horizontal Curves

The maximum design speeds for various horizontal curve radii are as summarized in Table 16-15.

Table 16-15: Design Speed vs Radii

Maximum Design Speed	Radii
30 km/h	30m but <55m
40 km/h	55m but < 90m
50 km/h	Radius > 90m

These design speeds were applied to curves with the corresponding radii along each haul route.

16.9.1.2 Vertical Curves:

The travel speeds on various gradients were calculated by first segmenting the route at points with major changes in road gradient. A rolling resistance of 2.5% was then added to these gradients to establish the total resistance. Using the rim-pull curves provided on the manufacturer's website, the travel speeds for the corresponding resistance were extracted for empty and loaded conditions.

16.9.1.3 Total cycle

The total travel time (roundtrip travel time) was estimated by summing the loaded travel time with the unloaded travel time and adding time for loading, tipping and queuing. The cycle times used for variation routes from the pits to stockpiles or waste dumps is summarized in Table 16-16 and Table 16-17.

Table 16-16: Haul Truck Cycle Times from the Pits to Process Stockpile

Process Feed – Haul Truck Cycle Times					
Routes				Results	
Route	Haul road	Start	End	Total time	Productivity
				hrs	t/hr.
1	L100	Lucky	Process stockpile	0.66	121
2	L100+L400	EBR	Process stockpile	0.23	355
3	L500	WBR	Process stockpile	0.24	332
4	L100	Kokanee	Process stockpile	0.50	160
5	L100	Golden	Process stockpile	0.52	155
6	L600+L100	Bohemian	Process stockpile	0.66	121
7	L100+L300	Fosters	Process stockpile	0.38	209
8	L100+L600	Schooners	Process stockpile	0.74	109

Table 16-17: Haul Truck Cycle Times for Waste Rock from Pits to Waste Rock Dump

Waste					
Routes				Results	
Route No.	**Name**	**Start**	**End**	**Total time**	**Productivity [2]**
Units	None	None	None	hrs	m³/hr.
9	L100	Lucky	CWD-04	0.18	189
10	L100	Lucky	CWD-07	0.25	139
11	L100	Lucky	Golden	0.24	147
12	L100	Lucky	Kokanee	0.30	117
13	L400+L500	EBR	WBR	0.25	142
14	L500	WBR	CWD-01	0.21	168
15	L100+L300	Kokanee	Fosters	0.23	150
16	L100	Kokanee	CWD-03	0.16	212
17	L100	Golden	CWD-04	0.17	201
18	L100	Golden	CWD-07	0.14	248
19		Bohemian	Schooners	0.13	269
20	L600	Bohemian	L100	0.15	234
21	L100	Fosters	CWD-03	0.27	128
22	L100+L300	Fosters	Constr.[1]	0.34	102
23	L300	Fosters	L100	0.17	205
24	L600	Schooners	CWD-06	0.14	250
25	L100+L600	Schooners	CWD-07	0.27	127
26	L100+L600	Schooners	Constr.[1]	0.34	102
27	L100+L600	Schooners	CWD-04	0.18	188

[1] Waste rock from Fosters and Schooners is planned to be used for construction of haul roads as well as for rock fill for infrastructure

[2] 15% reduction in productivity has been included to account for working conditions such as traffic, worker productivity, road conditions and additional dump travel

16.9.2 Drilling and Blasting

Drilling and blasting will be required in the pits prior to loading. It is proposed that process feed is mined in 6 m benches, to allow for better selectivity, while waste rock is mined in 12 m benches. Drilling is proposed through use of a crawler mounted drilling rig, with 3 m drill steels. A powder factor of 0.3 kg/tonne is proposed for process feed to allow for greater fragmentation whereas a powder factor of 0.25 is proposed for waste rock, which is considered adequate for loading. The burden and spacing is proposed to vary between 3 and 3.6 m to 3.5 and 4 m. Blasting is proposed to be done using ANFO which will be stored on site in ANFO silos, using nonel tubes for initiation.

Blast hole sampling is proposed as a means of grade control and delineation of process feed and waste rock prior to loading. A bulk explosive truck has been provided for transport of ANFO to blasting locations.

16.9.3 Loading and Hauling

Loading will be done using front end loaders and shovels. It is proposed that waste rock is loaded using front end loaders and process feed using shovels. Hauling of process feed is proposed using 80 t long distance haul trucks based on the Haulmax™ brand, these trucks have CAT™ parts but have been modified for long distance hauling. The trucks are also narrower (4.7 m wide) which allows for narrower haul roads within the pits. Waste rock haul is planned using 65 tonne, 40 m³, rigid frame dump trucks.

16.9.4 Support equipment

Support equipment planned includes dozers for doing drill pad preparation, haul road construction and for levelling waste rock on waste rock dumps. Dozing will also be done on the HLF both for stacking of material on the heap leach into cells and for ripping and heaping material for loading when mining the old heap leach material.

Provision has been made for a pit services loader, which will assist with pit work as necessary. The pit services loader is planned as a smaller loader, which will also undertake loading of material from the old heap leach.

Maintenance equipment has been provided for in the form of fuel and lube trucks, a welding truck, a truck with crane for general maintenance, a tire handler for large truck and loader tire handling and LDVs for maintenance, management and support staff.

A water truck is planned for dust allaying along haul roads. A lowboy truck has been provided for transporting the drills, dozers and shovel between pits, as multiple pits will be mined concurrently at times.

A mobile crusher, a sheep’s foot compactor and a grader have been provided for haul road construction and maintenance over the life of mine.

16.10 Drainage and Dewatering, and Electrical Services

Pit dewatering pumps have been included in capital and operating costs for the PEA. This is included as diesel powered units, using lay flat hose to pump water out of the pits. Water diversions will be planned around the pits to limit run-off ingress into the pits and pit water will be tested prior to disposal. No electrical supply is planned for the mining, as all equipment is planned to be diesel powered. However, two lighting plants will be provided for additional illumination where required during mining. These will be self-contained diesel or gasoline powered units.

16.11 Mining Equipment, Labour and Consumables Schedule

Table 16-18 shows the estimated hours used to generate operating costs for significant equipment over the life of mine.

Table 16-18: Mining Related Equipment Hours, Consumable Quantities and Labour Numbers Over Life of Mine

	Total/ Max	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Machine hours											
Blast hole drills ore	11,489	-	-	1,820	940	1,943	1,634	1,304	812	1,708	1,328
Blast hole drills waste	35,776		5,422	3,186	4,749	3,797	3,794	4,076	3,971	4,295	2,486
Shovel	12,082	-	-	1,914	989	2,043	1,718	1,371	854	1,796	1,397
Front end loader large	37,868	-	5,690	3,281	5,014	3,894	3,974	3,996	5,113	4,395	2,510
Front end loader small	70,346	-	2,659	9,409	9,516	9,685	9,099	8,019	8,598	7,534	5,827

Table 16-18: Mining Related Equipment Hours, Consumable Quantities and Labour Numbers Over Life of Mine

	Total/ Max	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Machine hours											
Dump trucks 80 tonne	70,467	-	-	12,362	5,366	12,955	9,239	4,660	3,772	12,611	9,501
Dump trucks 65 tonne	126,798	-	19,524	11,044	15,361	7,255	13,775	13,277	22,260	15,622	8,678
Dump trucks 40 tonne	72,529		3,978	11,234	11,490	11,490	10,530	6,350	6,346	6,348	4,763
Bulldozers	212,12	967	2,978	2,148	2,812	2,107	2,948	1,669	2,513	1,703	1,367
Grader	4,832	-	166	645	658	665	678	656	633	633	99
Compactor	3,188	-	663	323	329	332	449	409	316	316	50
Mobile crusher	6,746	-	373	1,656	1,656	1,656	402	362	310	310	22
Consumables											
Total diesel used Mltrs	28.36	0.04	2.73	3.42	3.38	3.35	3.44	2.85	3.35	3.48	2.32
Explosive tonnes	13,157		1,581	1,359	1,604	1,567	1,501	1,508	1,339	1,658	1,041
Labour numbers											
Mining operators	33	1	26	33	33	32	33	27	32	33	23
Maintenance staff	25	1	21	24	25	23	24	21	24	24	17
Management and support	16	1	12	16	16	16	16	16	16	16	16
Total mining staff included in mining costs	74	3	59	73	74	71	73	64	72	73	56

17.0 RECOVERY METHODS

17.1 Heap Leaching

The existing heap leach pad consists of 7 cells that will be reprocessed to extract gold that remains from previous operations. As this material is removed, fresh mineralized material will be placed onto the existing cells, as well as 3 new leach cells (Cells # 8, 9 and 10).

17.1.1 Engineering Design Criteria

The following Table 17-1 describes the design criteria used in developing the new leach pad cells.

Table 17-1: Heap Leach Pad Design Criteria

Item	Quantity/Criteria
Mine Life	▪ 8 years
Life of mine (LOM) ore quantity to be stacked on heap leach pad	▪ 13.5 M tonnes ▪ 10.3 M tonnes of new ore

Table 17-1: Heap Leach Pad Design Criteria

Item	Quantity/Criteria
	▪ 3.2 M tonnes from old heap
Ore Production	▪ Nominal 7,500 tonnes per day (tpd) (client information) ▪ Design 8,625 tpd (client information)
Ore Angle of Repose	▪ 35˚ (1.4H:1V) (assumed)
Final crush size (P_{80})	▪ 9.5 mm (calculated by SGS)
Ore geotechnical parameters	▪ Angle of Internal Friction (Phi) =35°, Cohesion (c) = 0 kPa
Leach pad type	▪ Permanent, multiple lifts
Initial stacking capacity	▪ Minimum of 1 cell non-winter season
Stacking schedule	▪ 230 days per year (client information)
Stacking Rate	▪ Nominal 7,500 tpd (client information) ▪ Design 8,625 tpd (client information)
Agglomeration	▪ Lime and cement (SGS)
Stacking method	▪ Truck tipping and dozer Spreading of Ore (client information)
Stacked dry density of ore	▪ 1.6 t/m³ (based on Amec loading test results, AMEC 2012)
Stack / lift height	▪ 8 m lifts, max heap height 40 m; ▪ first lift will be 7.0 m (based 7,500 tpd and 230 days of stacking per year) ▪ New cells will have at least 1.5 m of ore and 0.5 m of overliner material to cover the entire three cells before the winter to protect the liner and piping systems against freezing.
Overall slope angle of stacked ore	▪ 2H:1V, 26.6˚
Coefficient of permeability of stacked ore	▪ 0.005 cm/s (typical) (taken from Amec test results report using a load equivalent of 15 m, AMEC 2012)
Ore moisture contents	▪ ROM ore (dry weight basis) 3.09% nominal and design (calculated by SGS) ▪ Agglomerated ore 6.38% nominal, 8.7% design (calculated by SGS) ▪ Leached ore 11.22% nominal and design
Leach Pad	**Quantity/Criteria**
Arrangement	▪ Single phase pad with three additional cells ▪ Cells 8, 9 and 10 total lined area is 173,771 m² ▪ Old Leach Pad area (cells 1 – 7) is 305,012 m²
Grade	▪ Grade existing ground is 11 percent to the south ▪ 2.0 percent grade to the west parallel to the long axis of the cells
Containment Dike	▪ Constructed along the south and west edges of the leach pad to provide structural stability ▪ Constructed of well graded rock with a maximum particle size of 600 mm, less than 50% finer than 25 mm, and less than 10% finer than 0.075 mm ▪ 3H:1V slope on the uphill side to allow liner placement ▪ 2H:1V on the downhill side ▪ 2 m vertical distance between the toe of the heap and the top of the dike to convey spring runoff. ▪ Exiting dike will be removed and a new one will be constructed founded on bedrock to provide additional measure of stability.
Cell Divider `	▪ 2mm (80-mil) LLDPE Geomembrane FLAP @1.5H:1V within the overliner
Solution Application	**Quantity/Criteria**

Table 17-1: Heap Leach Pad Design Criteria

Item	Quantity/Criteria
Active Leach Surface	▪ 35,156 m² nominal ▪ 40,430 m² design
Leach Schedule	▪ 365 days per year (client information)
Leach cycle	▪ Total extraction time 60 days ▪ 30 days for primary extraction (assumed by SGS) ▪ 30 days for secondary extraction (assumed by SGS)
Solution application method	▪ Drip emitters and sprays for side slopes (buried during cold weather operations) (provided by SGS)
Solution application rate	▪ 12 l/hr./m² (provide by SGS)
Flow rate	▪ 422 m³/hr. nominal (calculated by SGS) ▪ 485 m³/hr. design (calculated by SGS)
Seismicity	**Quantity/Criteria**
Design Basis Earthquake (DBE)	▪ PGA= 0.143g (1 in 475 yr. return period; 2010 National Building Code)
Maximum Credible Earthquake (MCE)	▪ PGA= 0.250g (1 in 2,475 yr. return period; 2010 National Building Code)
Geotechnical Stability	**Quantity/Criteria**
Minimum embankment Factor of Safety	▪ Static Loading – 1.3 (short term), 1.5 (long term) ▪ Seismic Loading 1.0 (pseudostatic)
Permafrost	▪ Permafrost encountered in the pad or pond foundations, if thawed or unstable, will be removed
Solution Collection Ditch	**Quantity/Criteria**
General	▪ Outside the leach pad along the pad toe (western edge)
Design	▪ Houses conveyance HDPE SDR 9 pipes for three solution collection systems; pregnant solution, intermediate or low grade solution and leak detection solution. Solution will report to the solution tank and pump system located in the ADR plant.
Depth	▪ 1.0 m
Bottom Width	▪ 3 m
Slopes	▪ 2H:1V
Grade	▪ 11 percent
Liner System	▪ 0.15-m thick granular pipe bedding over a geotextile over a 2-mm (80-mil) HDPE geomembrane. Geomembrane anchor trench at 0.4-m minimum depth.
Groundwater	**Quantity/Criteria**
General	▪ Water table at 49 to 90 m below the ground surface (Feasibility Report Vol. II, Rescan Engineering, 1995) ▪ No drainage system will be required beneath the liner system. ▪ Note, unforeseen seepage may be encountered during construction, for which additional measures may be required.
Pad Liner System	**Quantity/Criteria**
Ore Cushion with Solution Recovery Piping	▪ 0.5 m of crushed and screened ore to protect the lining system from damage by ore placement.
Geosynthetic Liner	▪ 2.0 mm LLDPE geomembrane liner.
Soil liner	▪ 0.3 m of soil liner consisting of on-site silty material.

Table 17-1: Heap Leach Pad Design Criteria

Item	Quantity/Criteria
Subgrade	▪ Subgrade preparation to produce a firm and unyielding surface.
Leak Detection and Recovery System (LDRS)	▪ A system to collect leakage through the composite liner and convey it to monitoring points. The system to comprise drainage gravel and a network of drainage pipes to collect and convey any leaked solution.
LDRS monitoring	▪ Monitoring of the flow into the LDRS to ensure that allowable rates (determined by permitting authorities) are not exceeded. Mitigation procedures to be defined should rates be exceeded.
Frost protection	▪ Liner to be protected from seasonal frost penetration by maintaining a minimum of 3 m of dry ore above the cushion layer.
Pad Solution Drainage System	**Quantity/Criteria**
General	▪ The drainage system reduces hydraulic head on the liner and speeds solution recovery. It consists of a free-draining cover fill layer supplemented by drain pipes.
Liner Cover Fill	▪ 0.5-m minimum loose lift thickness of minus 25-mm free-draining granular material with less than 5 percent passing the No. 200 ASTM sieve size (may be crushed ore). ▪ Increase layer thickness to 1 m above larger diameter primary collection pipes. ▪ Placed in a single lift with no compaction ▪ Permeability at $1x10^{-02}$ cm/sec or higher
Solution Drain Pipes	▪ 100 mm diameter perforated corrugated polyethylene (PE) secondary collection pipes placed diagonally across the pad cell on 6-m centers header pipes ▪ collection pipes placed diagonally across the pad cell on 6-m centers header pipes ▪ 375 mm diameter perforated corrugated PE header collection pipes placed in the downhill direction within the cell interior and along the cell divider FLAP ▪ 400 mm diameter corrugated PE conveyance pipe contained in the collection ditch (two pipes: pregnant and intermediate solution) ▪ 600 mm diameter corrugated PE conveyance pipe contained in the collection ditch (leak detection system).
Drain Pipes Capacity	▪ 150% Solution flow plus the 100-year/24-hour storm flow of 37 mm
Collection Pond(s)	Quantity/Criteria
General	▪ Diversion berms will be constructed uphill of the pad to divert storm runoff from uphill catchments into natural drainages. ▪ A diversion ditch will be constructed along the west side at the toe of the containment dike
Alignment	▪ North and east diversion berms with alignments to be finalized in the field to suit ground conditions and minimize cut and fill
Design	▪ 2H:1V side slope ▪ Capacity: peak flows from the 100-year/24-hour storm of 37 mm from upstream catchments ▪ Freeboard 0.3 m

17.1.2 Heap Pad Slope Stability

The leach pad slope stability analyses included an evaluation of the planned foundation, pad liner, ore heap conditions, and containment dike. The stability analyses considered maximizing the crushed ore tonnage for construction and operation with stable stacked heap slopes on the pad liner system. Final spent and rinsed heap slopes for closure may require some slope flattening for long term erosion and re-vegetation conditions, but closure slopes were not considered for this PEA.

The HLF was evaluated for both static and pseudo-static (earthquake) conditions using a peak ground acceleration (PGA) with a return period of 475 years (10% exceedance in 50 years) and a 50 percent horizontal ground acceleration factor for the analyses. The engineering design criteria are presented in Table 17-2 and provides for an operational minimum static factor of safety of 1.3 for the ore heap. The minimum factor of safety for pseudo-static conditions is 1.0.

The assumed parameters for the ore heap slope stability analyses were developed from a data review of surface and subsurface conditions, planned construction, and past leach pad construction performance experience.

The planned ore heap limits were evaluated for both static and pseudo-static (earthquake) conditions using the geotechnical parameters presented in Table 17-2.

Table 17-2: Heap Leach Geotechnical Parameters

Material	Unit Weight (kN/m³)	Angle of Friction (˚)	Cohesion (kPa)
Ore	18.4	35	0
Clay liner interface	17.7	18	0
GCL Interface	17.7	13	0
Dike	19.0	32	0
Weathered Bedrock	24.5	37	0

The slope stability analyses were performed using the computer program Slide 6.0 component of the Rocscience software package to conduct limit equilibrium slope stability calculations using the Morgenstern-Price general limit equilibrium (GLE) method, which satisfies both force and moment equilibrium. Analyses evaluated both smaller and larger composite circular surfaces as well as composite (block) failure surfaces with sliding along the underlying liner interface system, which typically control the stability of facilities of this nature.

The results of the stability analysis indicate that increasing the heap height from 30 to 40 meters with the existing dike geometry can obtain a factor of safety greater than 1.3 if cells 3 to 7 are re-processed, and if modifications to the downslope dikes of between 5.0 and 7.4 metres if cells 1 and 2 are reprocessed. The factors of safety under pseudo-static conditions are all greater than 1.0. The stability analysis includes a phreatic line with a maximum head of 2.0m on the pad liner and a water table near the ground surface below the liner.

Some surficial slumping or raveling of the individual angle-of-repose ore lift slopes may occur during storm runoff or earthquake events. Considering the low hazard nature of the fully drained granular ore heap fill structure and the low probability of an earthquake event at this site during the relatively short life of the mine project, the pad liner system should remain intact and the surficial slope erosion, if any, can be controlled by periodic maintenance around the perimeter of the heap limits.

While permafrost has been encountered in a number of historic test pits and borings, the surface extents within cells 8, 9, and 10 is estimated to be less than 5%. During the clearing and grubbing phase of the construction, occurrences of permafrost should be delineated and assessed on an individual basis. Mediation may require over-excavation and replacement with rock fill or constructing a rock fill blanket to insulate the area.

17.1.3 Settlement Assessment

A one-dimensional settlement assessment was performed to estimate the maximum settlement and the differential settlement at various locations in the foundation soil of the proposed heap leach pad expansion. The settlements were calculated for a maximum ore stacking height of 32 m, corresponding to 4 lifts of 8 m each.

The maximum allowable strain on the liner system is controlled by the strain tolerance of the LLDPE. The allowable yield strain for the proposed 2.0 mm LLDPE is 12 percent and the elongation at break is 250 percent. The settlement calculations show that the differential settlement on the foundation liner system caused by the ore material will be most critical where the distortion is maximum which is between the toe of the first lift and the crown of the first lift using soil profile TP-6. Settlement at the two points was calculated to be zero and 0.14 m respectively. The two points are 12 m apart. This differential settlement will produce an increase in the liner system length of 8.2×10^{-4} m, which is equivalent to a strain of 0.007 percent, which is lower than the allowable strain of 12 percent. Therefore, the liner system will not be damaged by differential settlement induced by the weight of the ore material.

17.1.4 Heap Pad Surface Water Management

The leach pad design includes a diversion ditch surface water run-on to the leach pad with gravity drainage along an existing access road in the east side of the heap. The ditch will be a V-shape channel with 2H:1V side slopes and with a minimum depth of 600 mm. The ditch will be connected to the existing ditch along Cells 1 to 7.

A diversion ditch will be constructed along the west side at the toe of the containment dike to keep dike runoff water from entering the solution collection ditch. Runoff from the heap surface will be collected by a ditch formed between the heap toe and the containment dike along the western edge of the heap.

17.1.5 Heap Leach Water Balance

A feasibility level water balance model was developed specifically for the HLF. The feasibility water balance model was expanded using the water balance model prepared by Viceroy Minerals Corporation as part of an Updated Solution Management Plan submitted to the Yukon Territory Water Board on December 1998 in regards to the Water Use License QZ96-007 for the Brewery Creek Mine.

Nine scenarios were developed to estimate monthly excess solution and freshwater make-up requirements; estimate solution volume that may potentially report to the pregnant, barren, and overflow ponds; and analyze the monthly storage capacity of the ponds during extreme events.

The leach pad water balance included available climatology data, estimated ore moisture and production conditions, and planned construction for the start-up leach pad and pond operations to a FS level. A spreadsheet computer model was used to model predictions, on a monthly basis, the average year water balance for storm pond sizing. Heap leach operations were simulated for average precipitation, wet and dry conditions to validate the process and event pond sizing and to estimate the monthly fresh water make-up requirements for the heap leach facility and for establishing the maximum process design flows.

The heap water balance was modeled on a monthly time step to account for an additional 13.5 million tonnes of ore to be placed on newly constructed heap leach pad cells 8, 9, and 10. Two years of rinsing (detoxification) is assumed to commence immediately after operations. These two years of rinsing and detoxification are included as part of the water balance. Rinsing is assumed to occur at the same rate as the solution application rate. The water balance model assumes that the entire pad (cells 8, 9 and 10) are fully constructed prior to placing ore on the pad.

The results show the yearly make-up water requirements and captured water for the Average Conditions and the 100-year Snowpack scenarios. The heap water balance model demonstrates that the heap will be a "net positive" balance that requires no make-up water requirements with the exception of start-up. The ponds will have sufficient storage capacity, on a monthly basis, this assumes that excess water generated is managed using solution management methods such as active evaporation, snow dozing, additional sprinklers and storing solution in inactive portions of the heap.

Based on the water balance results the maximum volumes that report to the Ponds under the nine scenarios are summarized in Table 17-3 below. The maximum volumes reported typically occur in April when the snowmelt occurs.

Table 17-3: Results of Analysis of Yearly Pond Maximum Volumes over duration of Operation

Scenario	Maximum Pond Volume
1	77,752
2	118,001
3	118,001
4	76,625
5	119,804
6	119,804
7	72,266
8	119,804
9	119,804

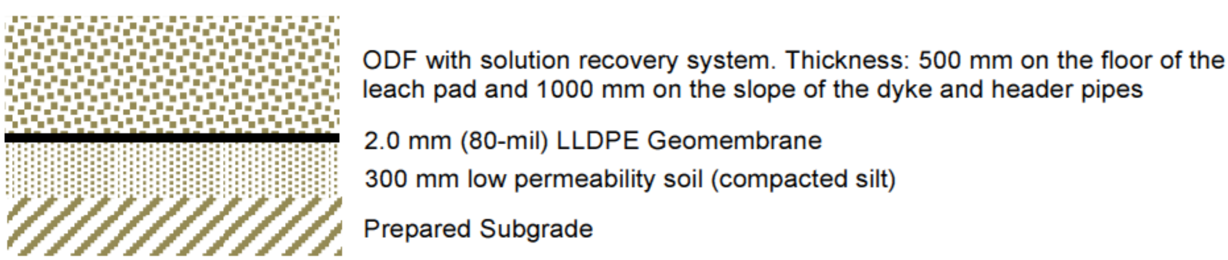
The water balance results indicate that the heap will be a net positive system, in that it will require minimal make-up water. The Ponds will have sufficient storage capacity (on a monthly basis) for all nine analyzed scenarios. This assumes that the excess water generated is managed using solution management methods such as active evaporation, snow dozing, using additional sprinklers, and storing solution in inactive portion of the heap.

The results of the water balance are limited to the information available and the assumptions used. The water balance can be revised and updated as more information becomes available.

17.1.6 Heap Leach Liner Design

The liner for the leach pad will consist of a geomembrane and underlying low-permeability bedding material, which is the state-of-practice liner system for heap leach facilities. The primary purpose of the composite liner system is to prevent the loss of HLF process solutions for both environmental and economic reasons. In addition to playing a role in preventing leakage, the underliner beneath the geomembrane is necessary as a transition layer between the geomembrane and the prepared foundation. The proposed liner configuration is shown in Figure 17–1.

Figure 17–1: Heap Leach Liner Layering



17.1.7 Containment Dyke West Side of Cells 8 to 10

A containment dyke will be constructed along the west edge of cells 8, 9, and 10 of the leach pad to provide structural stability for the ore heaps, and to control the solution draining from the heap. The dyke will tie-in with the existing dyke along cells 1-7. The dyke will be constructed of compacted rockfill, and will be lined on its uphill face with the leach pad liner system. The height of this dyke will vary as required to provide stability. A 5 metre wide road will be constructed along the crest to allow access during leach operations and construction. Slopes for the dyke will be 3:1 on the uphill side to allow for liner placement, and 2:1 on the downhill side. The 2 m vertical distance between the toe of the heap and the top of the dyke will allow for adequate containment of spring runoff and or heap slumpage.

The soil in the dyke foundation will be removed. The dyke will be founded on bedrock to provide an additional measure of stability.

17.1.8 Heap Leach Operations

Ore will be crushed for placement in the heap leach facility (HLF) at a nominal rate of 7,500 dry t/d. The HLF will be loaded using truck delivery and levelling using a bull dozer. Ore will be stacked at an overall slope of 2:1 (H:V) with benches between lifts. Each lift will be approximately 8 metres high. Loading will begin at the toe of cell 8 and progressively work upgradient and will alternate to permit sufficient time for leaching prior to placement of subsequent overlying lifts of ore. Drip emitters will be trenched into the top layer of the heap. Solution will be applied at an approximate irrigation rate of 422 m^3/hour over an approximate active leach area of 40,430 m^2 for a planned leaching cycle estimated at 60 days (30 days for primary extraction and 30 days for the secondary extraction).

Proposed heap leach operations will involve crushing and stacking 1.7 Mt of ore during 230 days per year with ore leaching year-round with three active leach areas being leached in the winter months. The ore placement will be ahead of the active leaching areas during summer operations, then the drip emitters will be buried at the start of winter for frost protection and leaching will continue to "catch up" with the ore placement.

The Brewery Creek site is located in the Yukon where winter air temperatures can reach minus 35˚C or colder in the winter months. A cursory review and comparison of heap leaching operations in cold climates and previous heap leaching operations at Brewery Creek indicate year-round leaching operations are considered achievable assuming design provisions are incorporated for adding and maintaining heat in the process solutions applied to the heap and proper operational provisions are incorporated. Operations in Alaska, northwest Canada, and Montana include Fort Knox in Fairbanks, Illinois Creek (reclaimed and closed in 2005) in Central Alaska, Brewery Creek in Central Yukon, and Beal Mountain Mine (closed in 1998) near Butte.

Solution heating considerations during previous operations at Brewery Creek utilized open process ponds, heat-traced and insulated barren tanks and pipelines, and buried emitters. Heap leach operations in this type of climate can be leached year-round with seasonal stacking (stockpiling and rehandling) due to the presence of frozen material on the heap leach during cold months. Frozen ore on a heap leach pad is generally detrimental to the operation due to the loss of percolation resulting in reduced recovery and possible heap instability from lateral solution flows to the heap slopes.

It is prudent to include the following provisions for seasonal stacking for pre-feasibility level studies:

- Sizing of the crushing operation and haul fleet to allow increased production rate during warm months;
- Sizing of the starter heap leach pad to accommodate more than 1 year of ore production to allow advanced stacking for at least the first winter season;

- Provision for ripping frozen ore prior to resuming leaching in the spring (a D-9 dozer with a single ripper may be required to break up frozen ore up to 2 m deep); and
- Provision for temporary over-irrigation to melt potential ice layers in the heap.

The process pumping system will include pumps, pipelines, valves, and associated controls to move solutions between the plant and the heap leach facilities. The process pumping and solution delivery systems should include necessary provisions for frost protection including:

- Heated barren solutions;
- Buried emitters (ripped in by dozer to 0.5m depth and covered with 2m additional ore is recommended for winter operations);
- Heat traced and insulated barren tank;
- Heat traced and insulated (or buried) pipelines as needed;
- Backup power supply to pumps via generators; and
- Provisions for pipeline drain down upon shutdown.

17.2 Recovery Methods

17.2.1 Process Plant Overview

Precious metals will be recovered from low grade mineralized material by heap leaching. To prepare the ore for leaching, a crushing plant is planned. The ore will be processed through the crushing plant at 7,500 tonnes per day, 230 days per year. Crushed ore will be placed on the heap by truck and leached with a dilute cyanide solution. Precious metals will be recovered from the leach solution by adsorption on activated carbon. The precious metals will be periodically desorbed from the activated carbon and the stripped carbon will be reactivated by an on-site kiln. Precious metals will be recovered from the strip solution by electrowinning. The product from electrowinning will be dried and smelted to produce dóre bullion which will be shipped from the site to refiners.

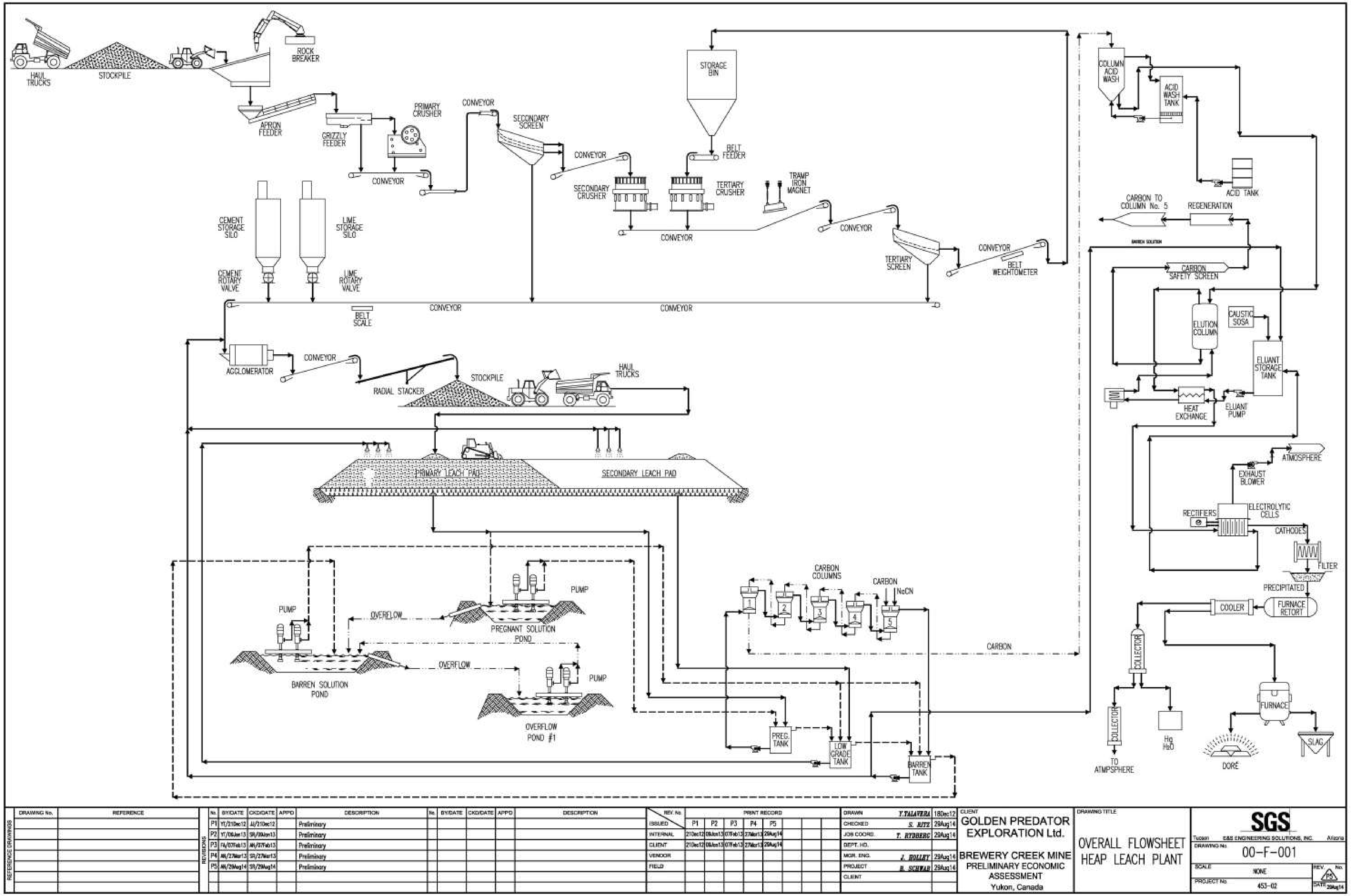
Based on the data provided by Brewery Creek, the following process plant flowsheet (Figure 17–2) has been selected:

1. Crushing Plant – Tertiary crushing, with primary jaw, secondary and tertiary cone crushers, and surge bin to feed agglomeration (Figure 17–3).
2. Agglomeration – Lime and cement are added to the main conveyor belt feeding the agglomeration drum. Agglomerates discharge the drum to form a crushed ore stockpile (Figure 17–4).
3. Ore Stacking – Truck Stacking.
4. Heap Leach Solution Management – Pumping and piping systems to circulate and collect leach liquors (Figure 17–5).
5. Carbon Columns – For precious metal adsorption (Figure 17–6).
6. Carbon Stripping and Refining – Concentration of gold solutions for electrowinning and production of final product (Figure 17–7).

7. Acid Washing and Carbon Reactivation – Carbon handling system designed to remove acid soluble deposits on the carbon surface and re-activate loading sites to maintain maximum gold loadings (Figure 17–8).

8. Final Detoxification – Solution detoxification for discharge (Figure 17–9).

Figure 17–2: Conceptual Process Plant Flowsheet



17.2.2 Crushing

A modular crushing system is to be purchased and operated by the Owner who incurs all the capital and associated operating costs. The conceptual Crushing Plant is shown in Figure 17–3.

The advantage of this option is that the Owner is in control of the output. The disadvantage is the capital expenditure ahead of production to procure and engineer the crushing system and lead times for modular equipment.

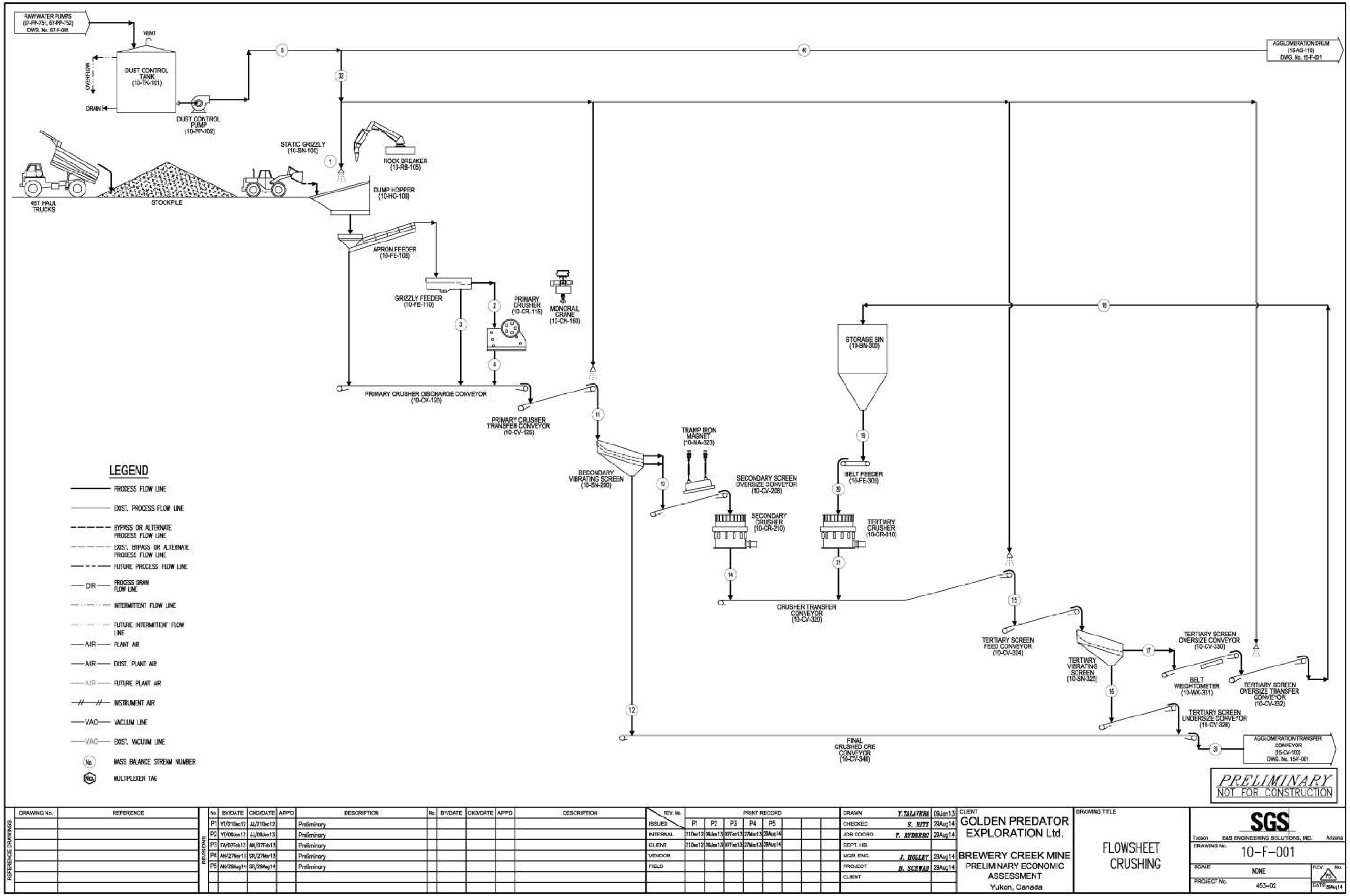
Run-of-mine (ROM) ore is delivered to the crushing plant from the pits by overland truck haulage. A large stockpile is located close to the primary crusher feed pocket. A static grizzly screen above the hopper limits the top size of rock fed to the crusher to 500 mm. Below the hopper, an apron feed transfers ore at a controlled rate to the vibrating grizzly screen.

Grizzly screen oversize feeds the primary jaw crusher. Grizzly screen undersize joins the crusher product on the primary crusher discharge conveyor which feeds the primary crusher transfer conveyor taking the ore to the second stage of crushing. A mobile rock breaker is available to service the crusher or ROM grizzly screen. The primary crushing circuit reduces the size of run-of-mine from a maximum of 500 mm to approximately 80 percent passing 90 mm.

Primary crusher discharge is fed to a double deck scalping screen to remove fines from the secondary crusher feed. Fines from this screen are finished product and discharge from the circuit to a small surge bin. Crushed product from the secondary crusher joins the tertiary crusher discharge and is conveyed to the tertiary screen feed bin. Water sprays are used for dust control.

Secondary and tertiary crusher discharge is fed to the tertiary screen by a conveyor. Screen oversize flows to the tertiary crusher feed bin. Screen undersize discharges to the crusher discharge conveyor. The tertiary feed bin is required to allow the tertiary crusher to operate in a choke fed condition. A conveyor feeds material from the bin into the crusher as needed to control the level in the crusher. Tertiary crusher discharge joins the secondary discharge and is recycled to the tertiary feed screen. Water sprays are used for dust control.

Figure 17–3: Conceptual Crushing Plant (SGS 67-F-001, 2014)



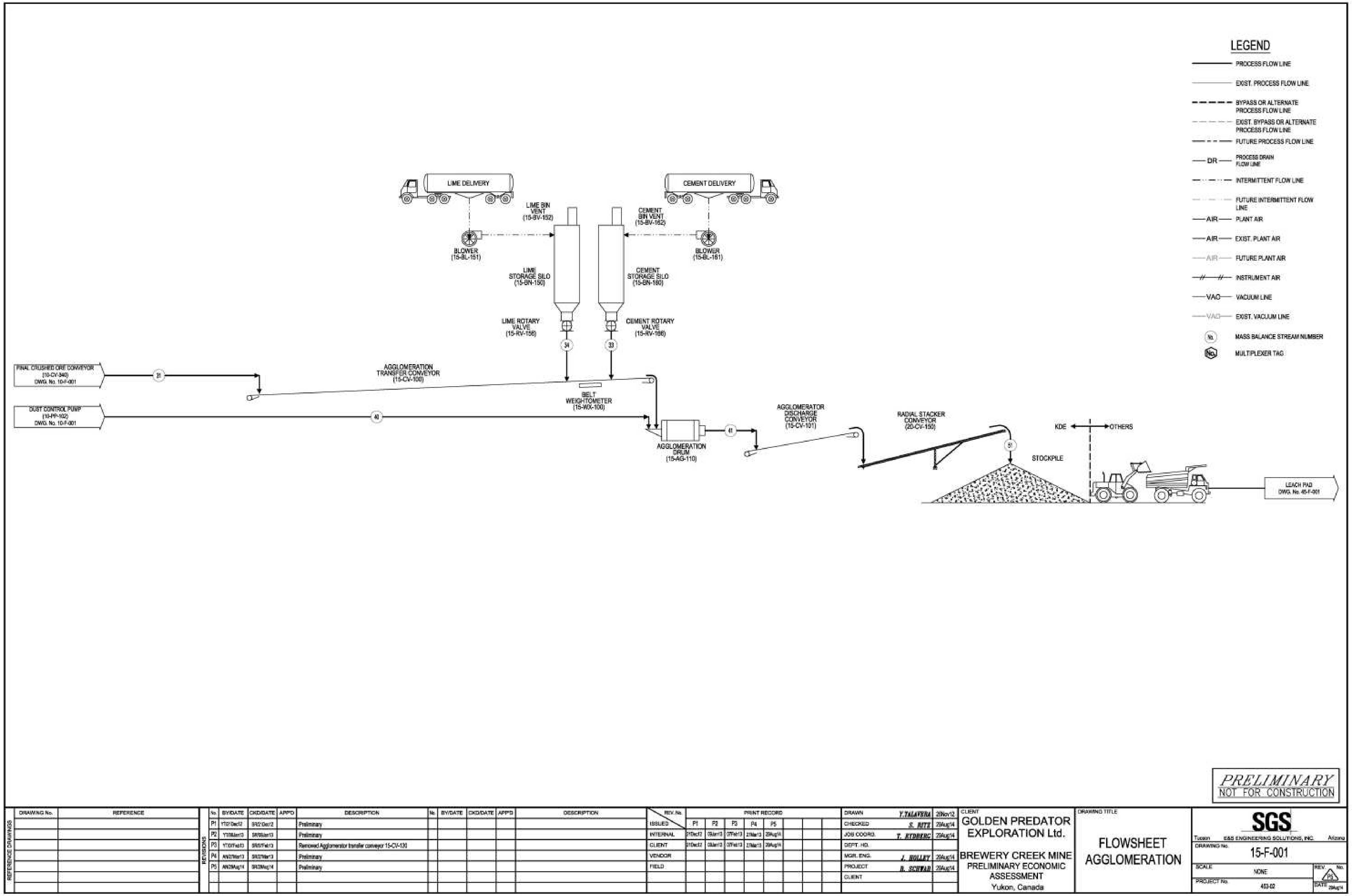
17.2.3 Heap Leach Stacking and Placement

Crushed ore at 100 percent passing 16 mm is conveyed to an agglomeration drum (Figure 17–4) where cement and lime are metered onto the conveyor belt using rotary valves. An agglomeration drum mixes the reagents with raw water to generate an agglomerated feed for the heap leach. The agglomerated material is stockpiled and then transferred to trucks using a front-end loader. The trucks place the agglomerates on the pad in accordance with the stacking plan.

Cement and lime additions are varied with the ore tonnage, ore type and ore placement in the heap based on input from the operations staff. Both reagents are trucked to the site, unloaded into on site silo storage, and metered from the silos to the leach feed conveyor by a rotary valve.

Cement and lime provide the necessary pH control for the heap leach operation. The amount of lime and cement added is designed to maintain the pH of the pregnant solution at or above 10.0. Additionally, cement is used as a binder for fine particles, increasing the permeability of the overall heap.

Figure 17–4: Conceptual Agglomeration Flowsheet (SGS 15-F-001, 2014)



17.2.4 Solution Management

Solution management at the Brewery Creek heap leach includes systems for collection, application, and evaporation of solutions. The complete leaching cycle for freshly mined ore is approximately 90 days or a total solution to ore ratio of 2 kl/tonne of ore. For reprocessed ore reclaimed from the original heap leach, the leaching cycle is 140 days or a solution to ore ratio of 3 kl/tonne. A conceptual flowsheet is seen in Figure 17–5.

During the primary leach cycle, low grade solutions are applied to the freshly placed ore. The low grade solution is collected from previously leached cells where small amounts of gold are still being extracted. This solution reports to the low grade solution tank, located inside the ADR facility, and is then pumped to the new ore cells. The resulting pregnant solution, or preg, is upgraded so that solution tenor to the ADR is maintained at the highest level possible.

Cyanide solution is added to both the low grade and barren solutions. Sodium hydroxide can also be added, if required, to maintain the solution pH above 10.5. Make-up water is added as needed to keep the solution flows in balance.

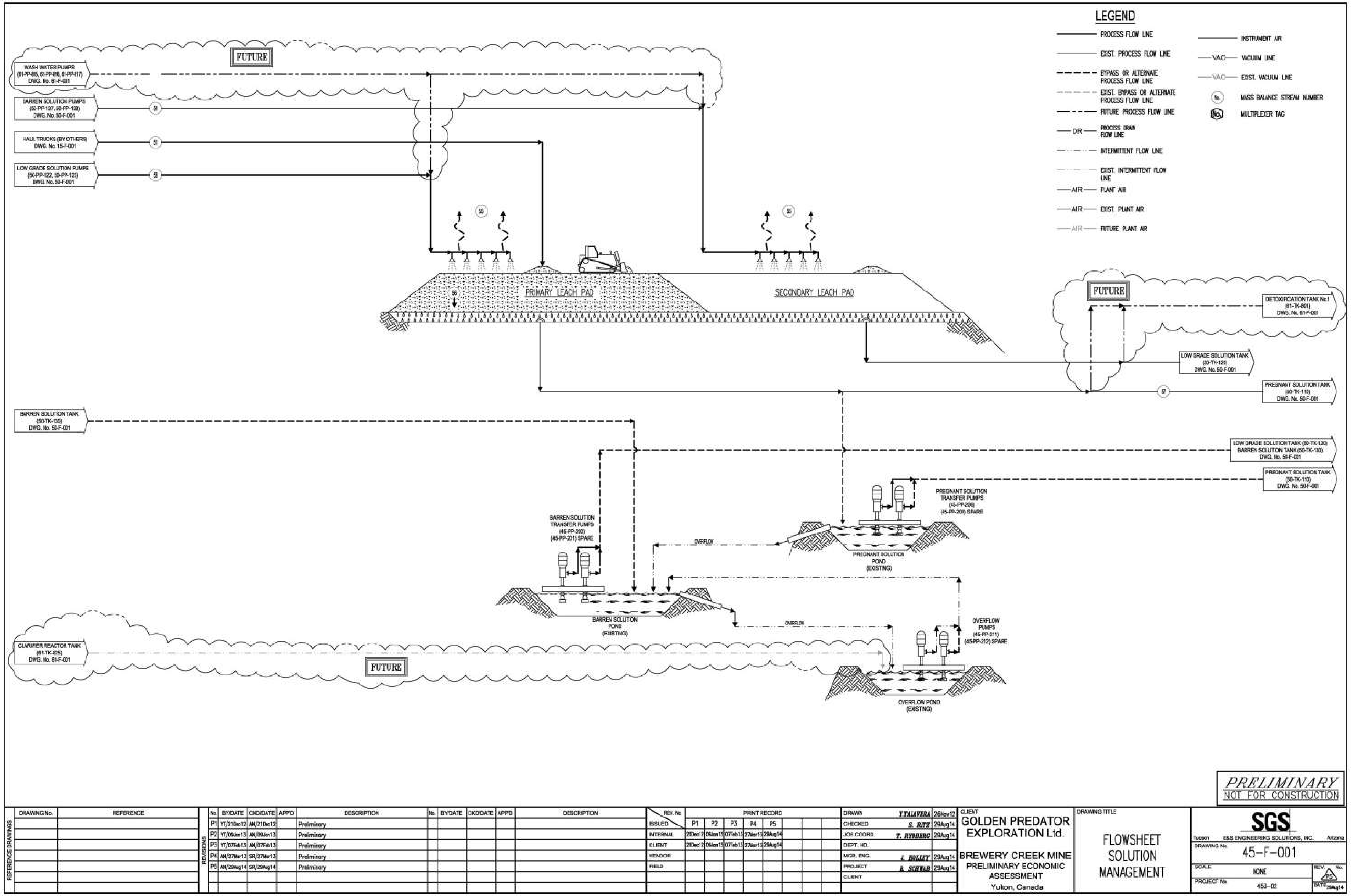
Solutions are applied to the ore using drip emitters at the rate of 12 $l/h/m^2$ of leach area. Water sprays or wobblers can also be used to encourage evaporation during those months when excess solution reports to the system, primarily during the spring snow melt, and to wet the pad side slopes.

The primary collection systems for the pregnant and barren solutions are the tanks located inside the ADR plant. Surge ponds are also provided maintain the overall solution balance. Both the low grade and barren solution tanks overflow to the barren solution pond. Pregnant solution tank overflows to the pregnant solution pond.

During the winter months, October to March, solutions continue to circulate through the ADR plant from the pad. Operators maintain the solution temperature by burying the emitter lines approximately 1-metre below the ore surface. Return lines are also buried and insulated and report directly to the ADR facility, which is a heated structure. Past operation at the Brewery Creek site has shown that this approach to solution management will maintain the solution temperatures year round.

Solutions can bypass the CIC columns so that carbon can be removed from the columns once solution tenor drops below a set target. Leaching continues to occur during the winter months and a skeleton crew is required to maintain critical equipment and monitor solutions.

Figure 17–5: Conceptual Solution Management Flowsheet (SGS 45-F-001, 2014)



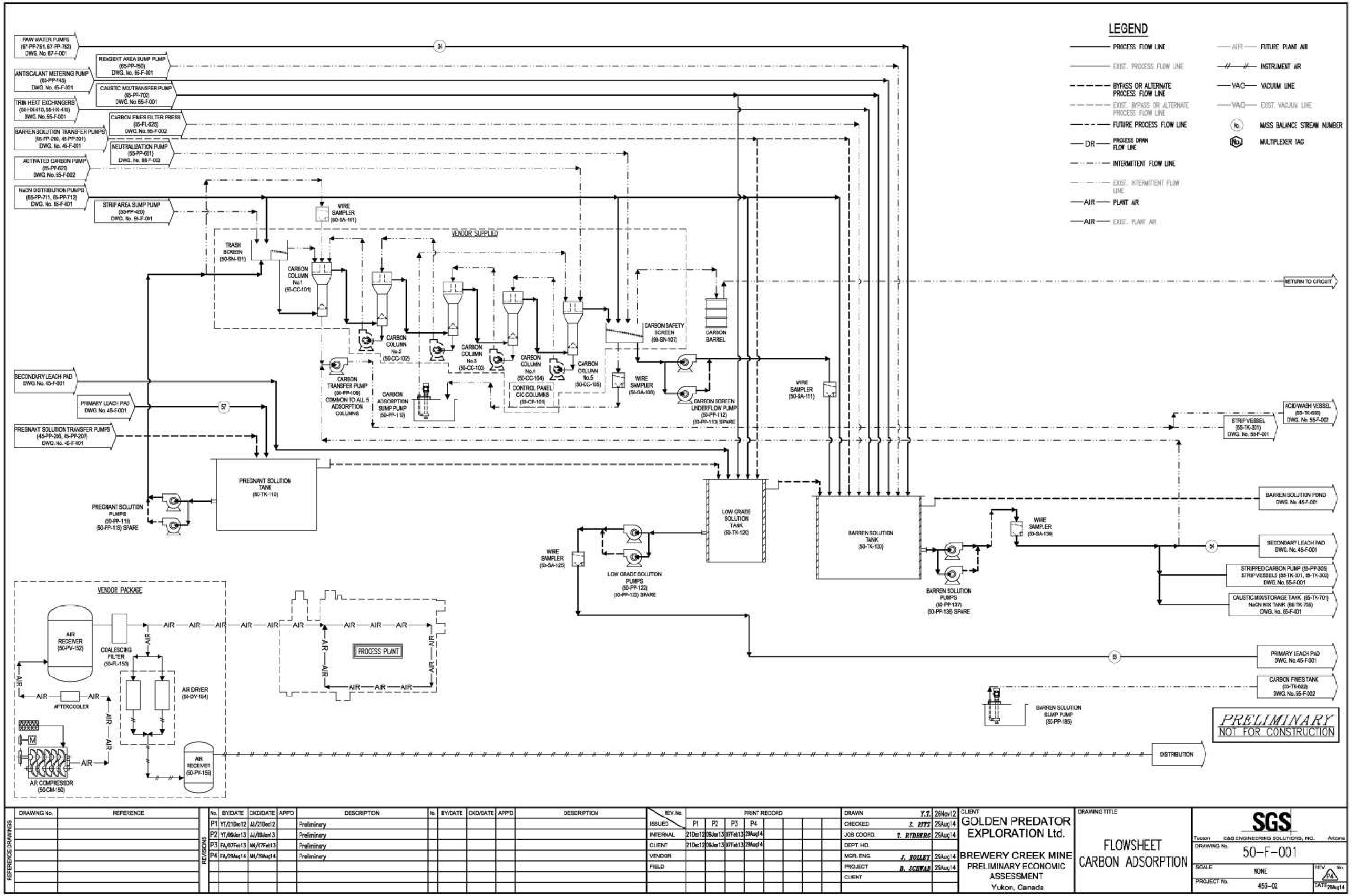
17.2.5 Carbon Adsorption

Pregnant solution is pumped to the carbon adsorption; or CIC carbon columns, from the preg solution tank. A single train of 5 columns is used to adsorb the gold from the solution. The final solution discharges over a safety screen to the barren solution tank. The conceptual Carbon Absorption Flowsheet is seen in Figure 17–6.

The train is stepped down to allow gravity flow of solution from stage to stage. Carbon is advanced counter currently through the columns using a recessed impeller pump mounted externally. Carbon pump suction and discharge can be controlled by automatic valves so one pump serves all CIC tanks in the train. Loaded carbon from the Number 1 column will be pumped to a dewatering screen which will discharge the carbon into the loaded carbon storage bin.

Barren solution from the Number 5 column flows over a vibrating safety screen. A small amount of solution is diverted to a wire sampler where the metallurgical sample is collected. The solution from the safety screen flows to the barren solution tank, where the cyanide strength is increased to the concentration required for leaching by metering in a 20 percent cyanide solution. Barren solution will be pumped through flow meters and returned to the leach solution distribution system through a pipeline.

Figure 17–6: Conceptual Carbon Absorption Flowsheet (SGS 50-F-001, 2014)



17.2.6 Stripping and Refining

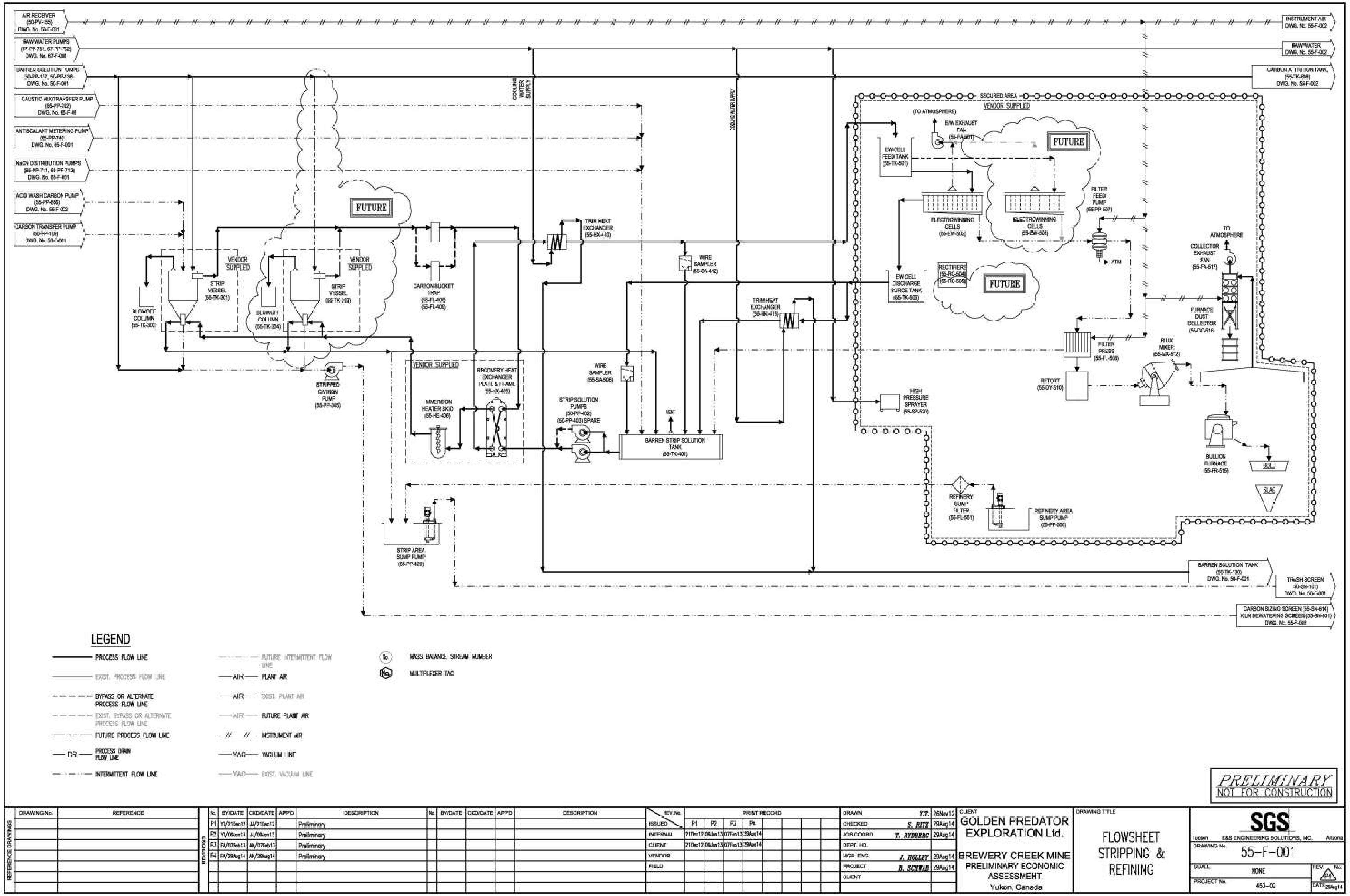
Gold on the loaded carbon is desorbed from the carbon surface using the Zadra process. The Zadra process operates at 150°C and 100 psig. Carbon is stripped using a hot caustic solution containing one to two percent sodium hydroxide and 0.2 percent sodium cyanide. The resulting elute solution contains high concentrations of precious metals sufficient for electrowinning. The conceptual Stripping and Refining Flowsheet is seen in Figure 17–7.

Pregnant solution from the elution vessel flows through heat exchangers where heat is recovered and used to preheat the incoming elution stream. Solution then flows through electrowinning cells where the gold is recovered from solution and deposited on to stainless steel mesh. The barren electrolyte then reports to the barren tank strip solution tank where it is reused. Solutions from the barren tank are pumped through the heat exchangers and into the elution column to close the loop. A small amount of bleed solution reports to the barren solution tank located at the end of the CIC train.

Two electrowinning cells are provided in the recovery circuit. One cell operates while one is cleaned. Gold and other metals are deposited on to stainless steel mesh cathodes. During the cleaning process, the cathodes are removed and the metal sludge is washed into a bin using a high pressure sprayer. Since the sludge handling and refining are batch operations, additional capacity can provided by running additional batches as required.

The resulting slurry is filtered and placed into a retort/dryer. The retort will collect any mercury that maybe present in the sludge. The dried metal cake is fluxed and smelted into dóre bars using an induction furnace. Off-gases are captured in a bag house dust collection system where precious metal dust is captured and returned to the system.

Figure 17–7: Conceptual Stripping and Refining Flowsheet (SGS 55-F-001, 2014)



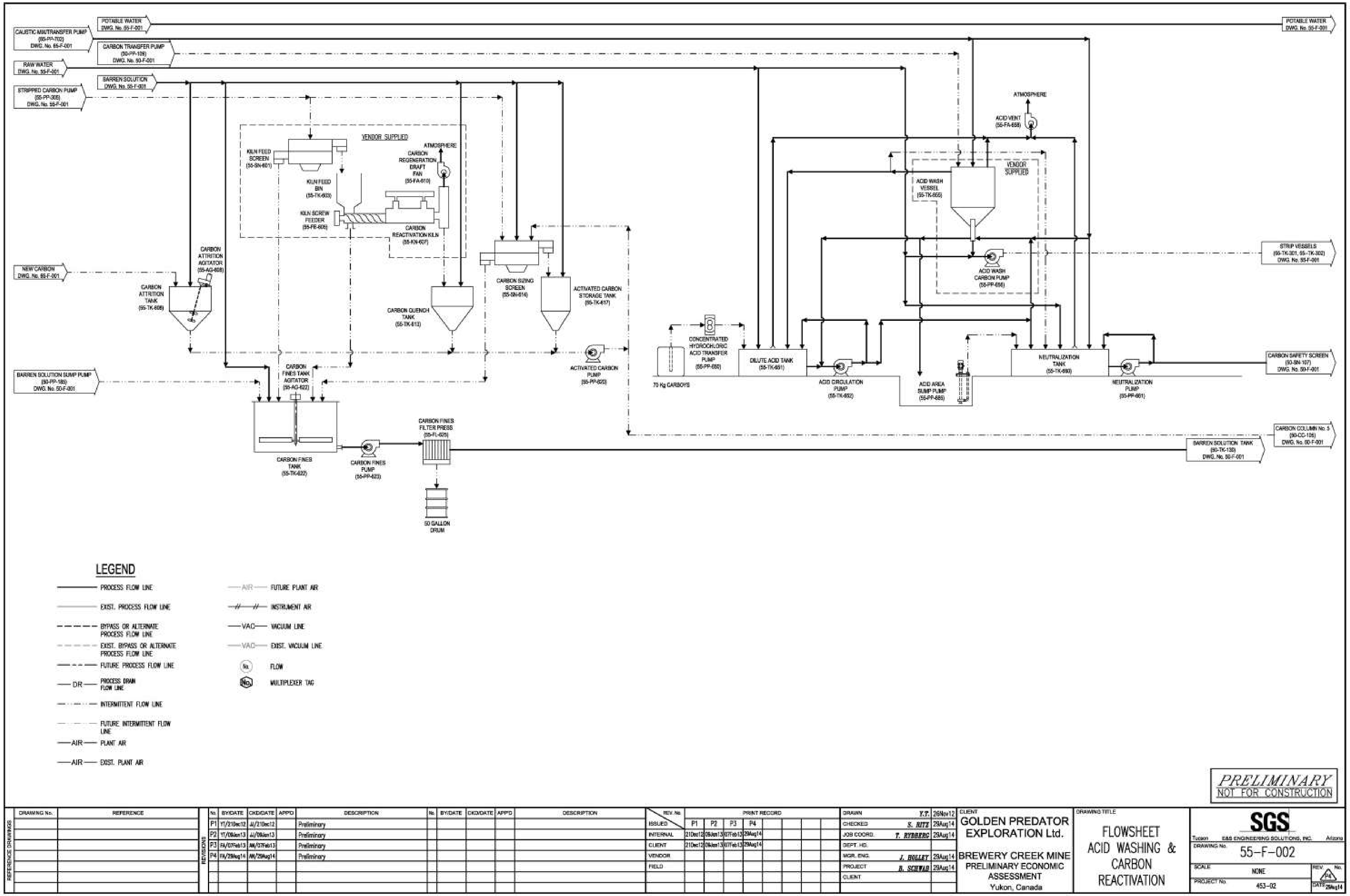
17.2.7 Acid Washing and Carbon Reactivation

To maintain acceptable carbon activity, the carbon must be acid washed. A dilute hydrochloric acid solution is used to remove scale from the carbon surfaces that result from contact with caustic solutions containing calcium. The carbon can be acid washed before or after the elution process. Once the wash cycle is complete, the carbon is rinsed with water and caustic to neutralize remove any residual acid. The conceptual Acid Washing and Carbon Reactivation Flowsheet is seen in Figure 17–8.

Before carbon is returned to the CIC process, some of it is thermally reactivated. One reactivation kiln is provided. The kiln operates in a reducing atmosphere to prevent the carbon from burning. Reactivation temperature of 600°C is applied to the carbon using and indirect heating method. The final product is quenched and reports to the activated carbon bin.

New carbon is added to the circuit through an attrition tank. Attrition removes the sharp edges and fines from the new carbon which can collect in barren solutions, plugging emitters. The attrited carbon reports to the activated carbon bin.

Figure 17–8: Conceptual Acid Washing and Carbon Reactivation Flowsheet (SGS 55-F-002, 2014)



17.2.8 Heap Detoxification (Future)

For this study, the chemical detoxification of the heap is based on the use of hydrogen peroxide for reasons of cost, simplicity and effectiveness. Hydrogen peroxide oxidizes free cyanide as follows:

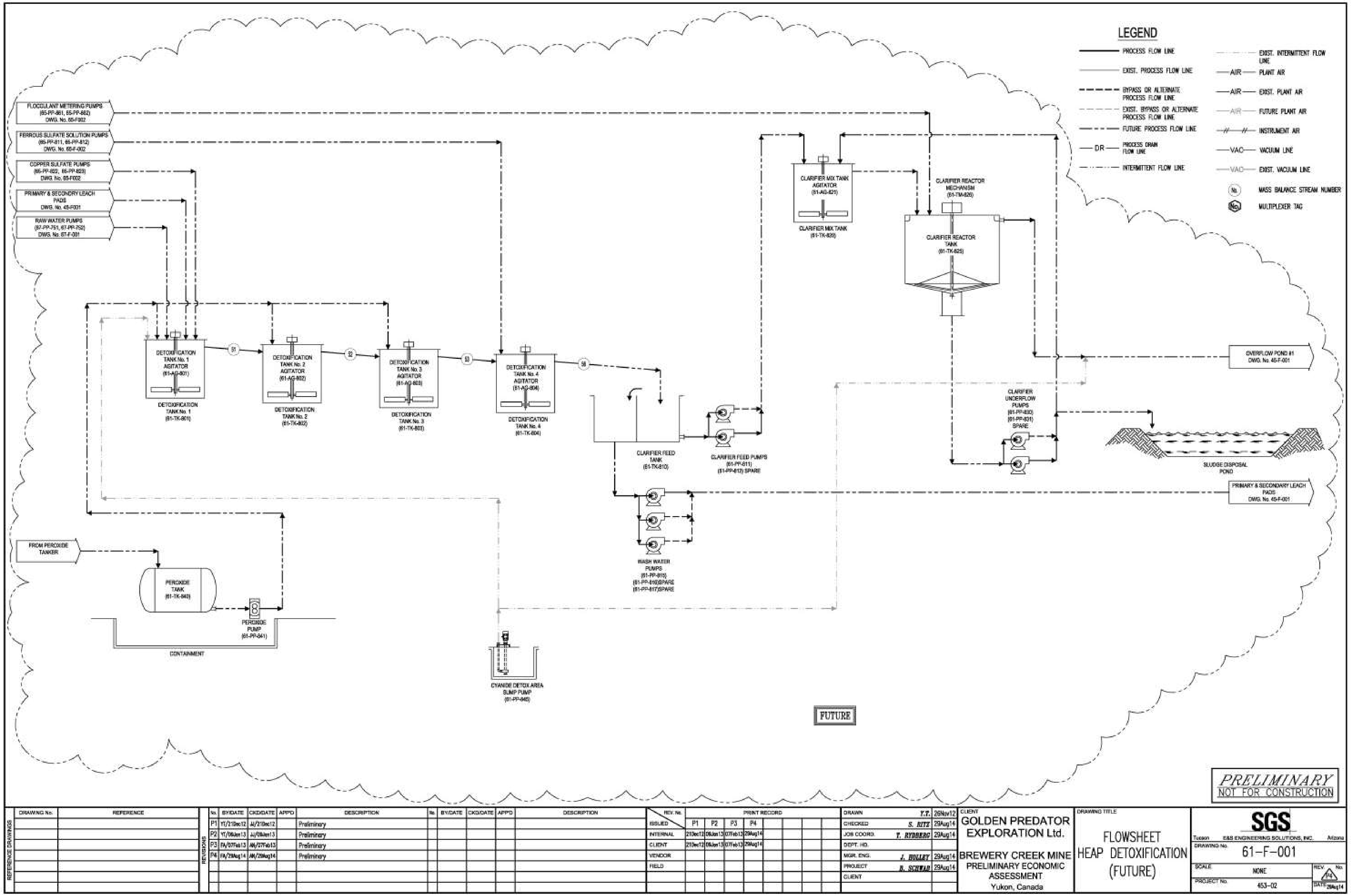
$$CN^- + H_2O_2 = CNO^- + H_2O$$

Other weak-acid dissociable (WAD) cyanide complexes are oxidized to cyanate and metal hydroxides. Cyanate complexes may undergo hydrolysis to ammonia and carbonate ions when the pH drops below 7.

$$CNO^- + 2H_2O = NH_4^{\ +} + CO_3^{\ 2-}$$

This system will be used in the future for remediation of the heap. The conceptual flowsheet is seen in Figure 17–9.

Figure 17–9: Conceptual Heap Detoxification Flowsheet (SGS 61-F-001, 2014)



17.2.9 Reagents

Reagents for the Brewery Creek heap leach consist mainly of cyanide, carbon, lime, cement, and sodium hydroxide or caustic. A small amount of acid is used in the carbon activation process as previously described.

Cyanide solutions are produced in batches from solid pellets or briquettes. Cyanide briquettes are dissolved using barren solution to create a 20 percent cyanide concentration. This batch is then transferred to a dosing tank where the solution is distributed.

Caustic solutions are also produced as batches from pellets. Fifty pound bags are used to create a 19 percent solution. This solution concentration is chosen as it is the highest concentration possible with the lowest freezing temperature, minus 28°C, eliminating the need for heating the solution.

17.2.10 Water

The raw water distribution system provides raw water for process requirements such as process water makeup. Raw water is taken from a basin at Laura Creek and pumped to a head tank for distribution to the process. A portion of the tank will be used to provide fire water storage for the system.

The potable water distribution system provides water for safety showers and other uses as required. A small head tank located inside the ADR plant facility and will be used to provide surge and head for the system. Safety water systems will be chlorinated and circulated to keep them clean and warm.

18.0 PROJECT INFRASTRUCTURE

For the purpose of the PEA, the required infrastructure has been based on complementing the layout used during historic operations and the exploration phase of the project.

The existing infrastructure on the property includes temporary camp facilities and other structural remnants of previous mining operations. The former maintenance shop was partially dismantled after closure of the historic mining operations and is currently being used as offices and exploration core logging facilities. Accommodations include mobile living quarters in the form of prefabricated trailers and temporary tent structures. There is capacity for approximately 45 people in these current facilities.

Existing support infrastructure includes:

- An all-weather access road approximately 20 km long, from the Dempster Highway;
- Incinerator;
- Sewage disposal system;
- Propane storage;
- Diesel and gasoline storage tanks;
- Core storage area;
- Weather station;
- Secure dumpster for waste material; and
- Power supplied by a diesel generator (150 kW) as the site does not have access to the Yukon power grid.

Table 18-1 below summarizes the current buildings used during the exploration phase.

Table 18-1: Current Buildings at Brewery Creek

Building	Size	Current Use	Potential Future Use
Mine dry	10 x 5 m	Dry for exploration drillers	Dry for office staff
Camp	6 Travco trailers	Exploration camp capacity for 45 people	Replaced by larger camp
Office and core shed	15 x 25 m	Site offices, core work, first aid room	Part of operation offices and ongoing exploration work.

18.1 Mine Site Infrastructure Required to Restart Brewery Creek mine

As the Brewery Creek mine is a past gold producer, construction of new mine site infrastructure will be based on previous operations. Historic mine records and drawings have been reviewed for consideration and planning of new infrastructure required for the development of the Brewery Creek property.

18.1.1 Planned Infrastructure Overview

Project facilities and infrastructure planned for Brewery Creek as part of the PEA include:

1. ROM storage and crushing facility;
2. An additional three heap leach cells to create a combined facility comprising 10 cells (seven of which are loaded), comprising a rock embankment, a lined storage area, pregnant, barren and overflow ponds, and leak detection and recovery system, solution recovery and monitoring systems;
3. Solution processing and adsorption desorption recovery processing plant (ADR facility, reagent storage, assay lab and laydown area;
4. Temporary stockpiles for recovered leach pad material;
5. Buildings and foundations, including administration and planning offices, accommodations, fuel storage, explosives and detonator storage;
6. Waste rock storage facilities;
7. Fresh water supply and conveyance systems including a pump house and piping to treat and distribute the water as process water, fire water, and potable water, diversion ditches for both contact and non-contact water;
8. Solid waste management facilities including an incinerator, a landfill, solid waste skips/bins and special waste storage facility;
9. Solid waste management facilities;
10. Site and access roads;
11. Power supply and distribution:
    a. a generator facility at the ADR;
    b. a substation; and

   c. power cables and lines;

12. Lighting, communication and control systems; and

13. Fire and emergency response facilities.

18.2 Planned Layout

Figure 18–1 through Figure 18–5 show the layout planned for Brewery Creek operations.

Figure 18–1: Brewery Creek Overall Site Layout

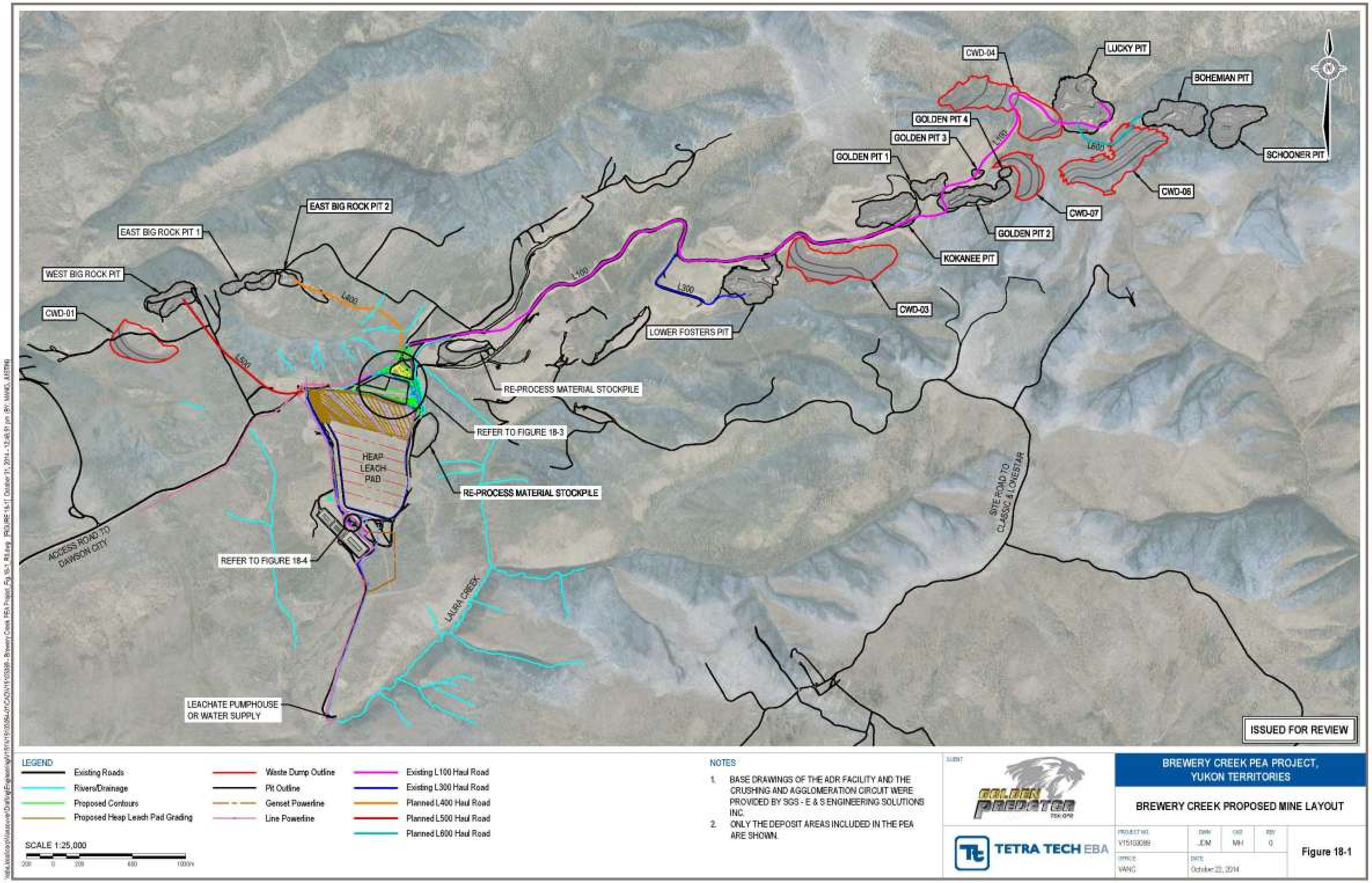


Figure 18–2: Brewery Creek Site Plan Showing Haul roads and pit locations

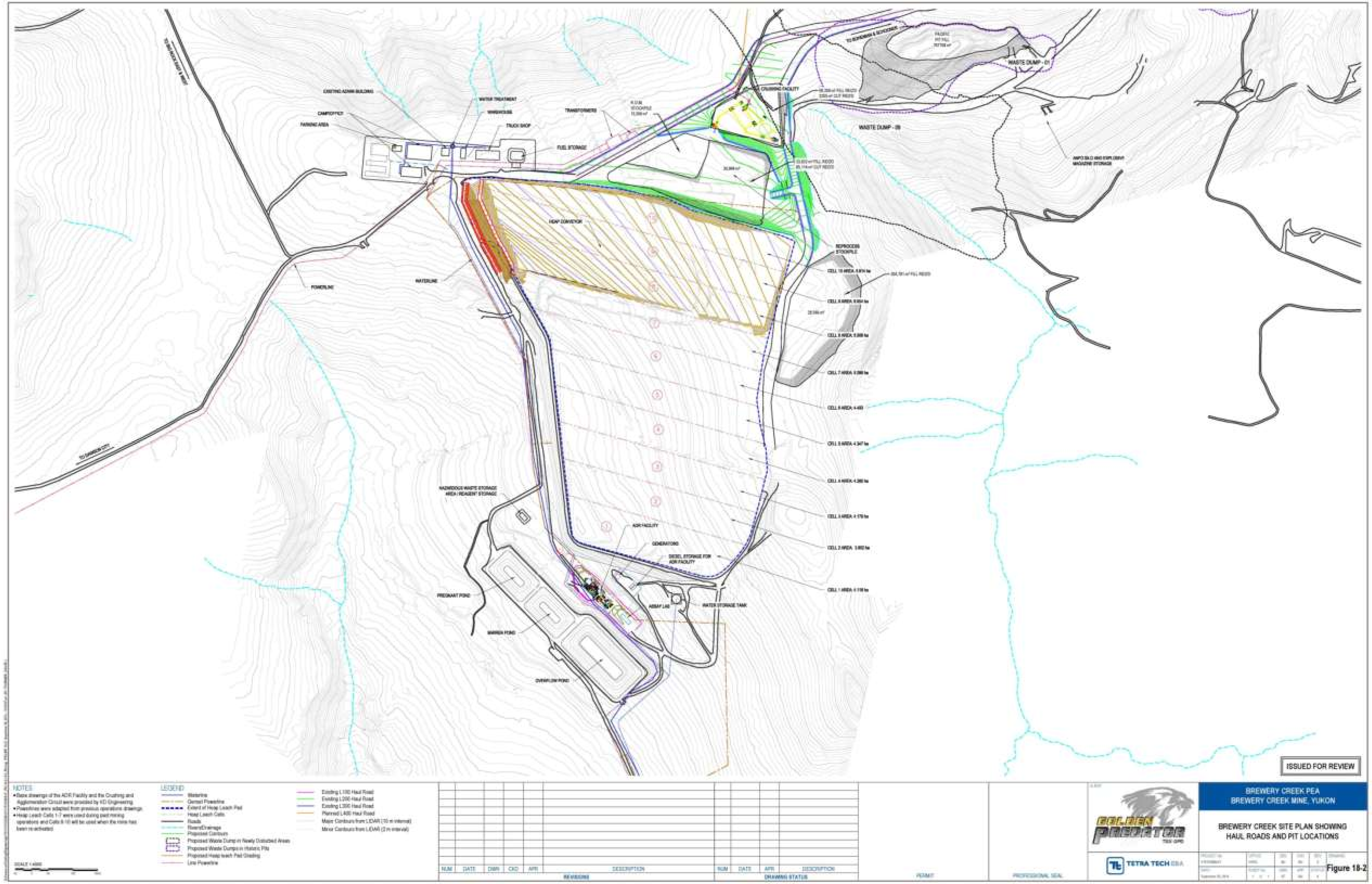


Figure 18–3: Brewery Creek Crushing facility

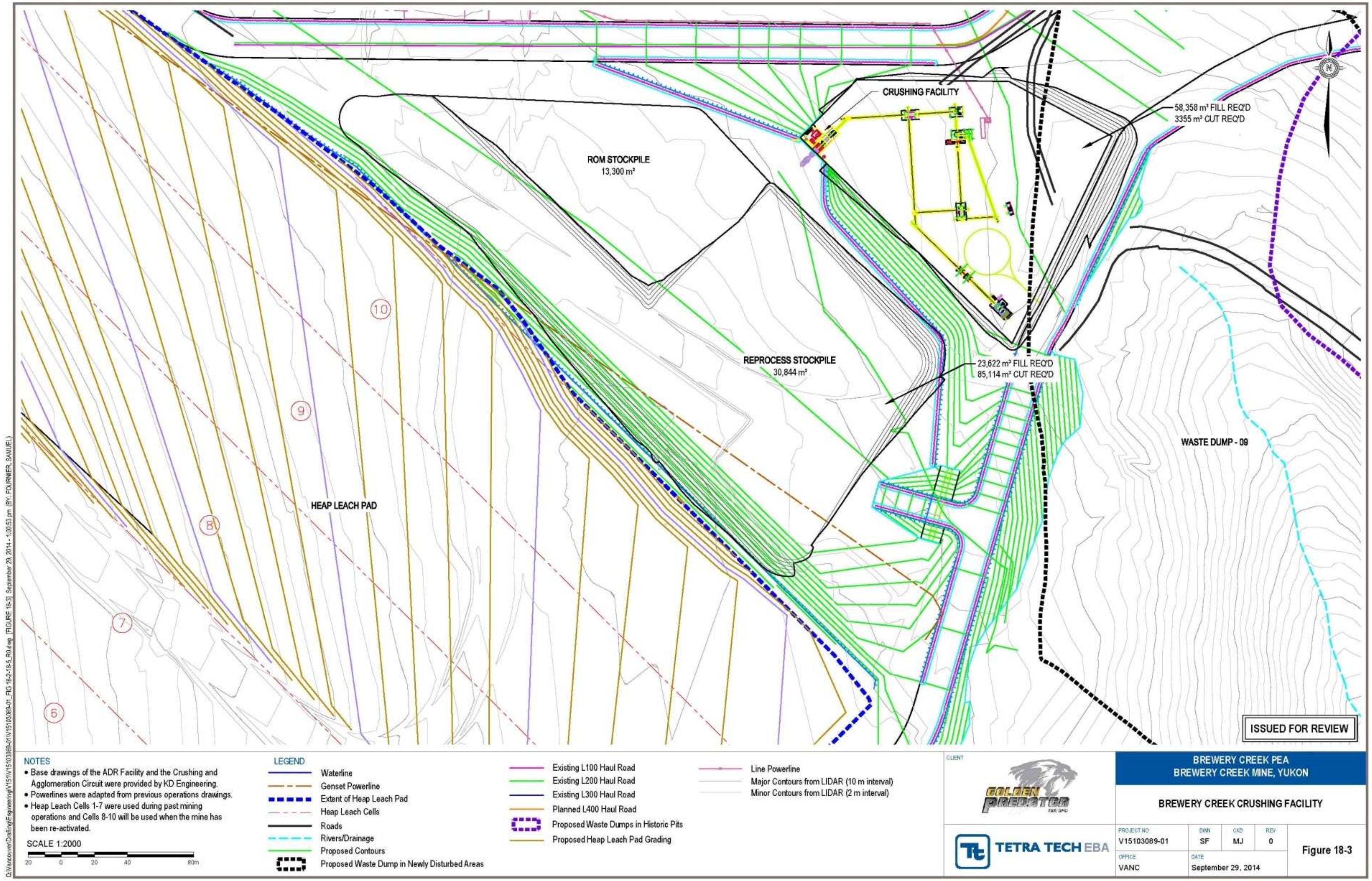


Figure 18–4: Brewery Creek Processing facilities

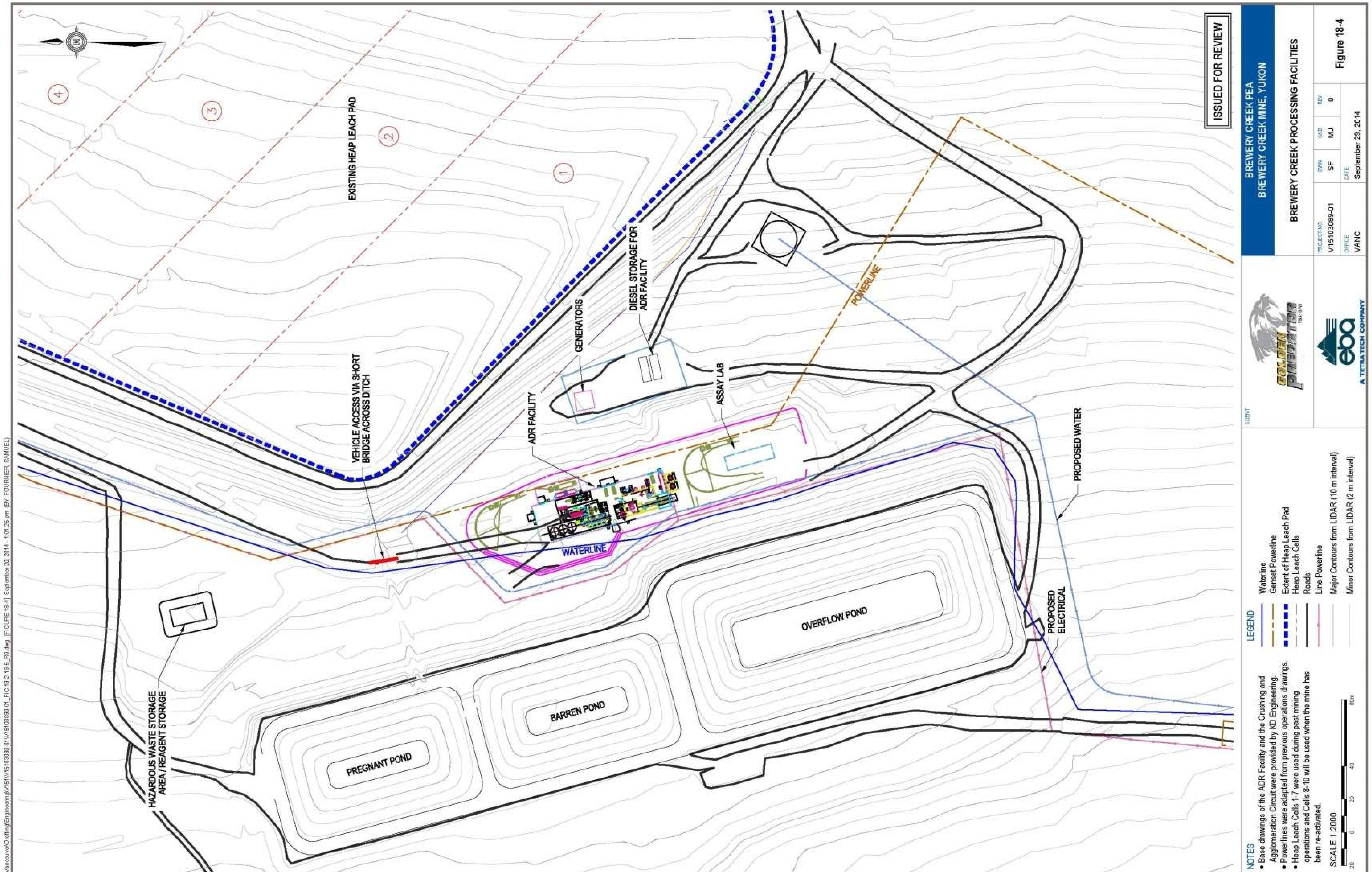
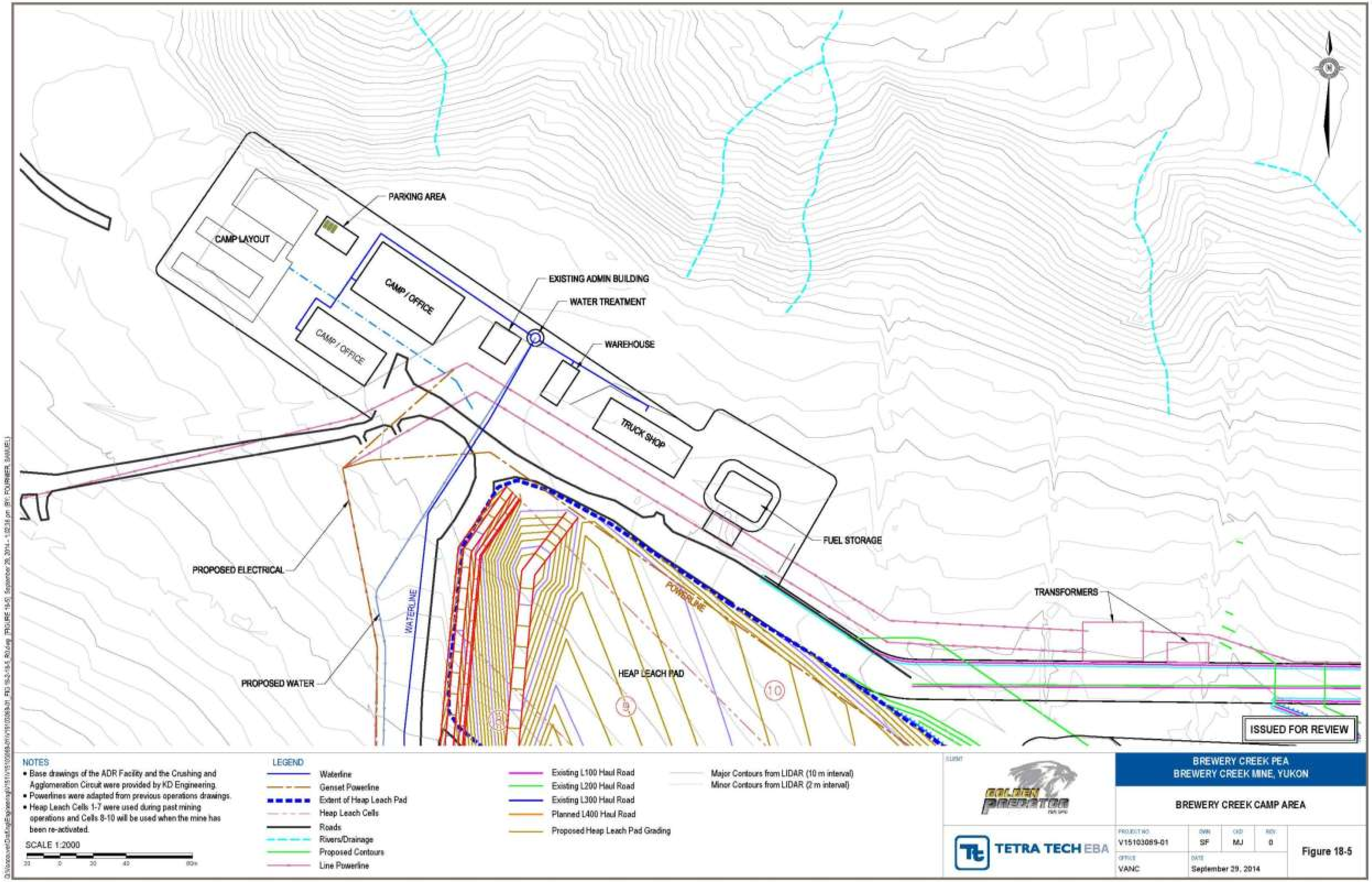


Figure 18–5: Brewery Creek Camp Area



18.3 ROM Storage and Crushing Facility

The capacity for the run-of-mine stockpile has been designed for 10 days of production at the mining rate of roughly 7,500 tonnes per day. Such a facility will allow for blending of higher and lower grade ores if required. The crushing facility will be located at the north east corner of the heap leach facility, so as to enable conveying of crushed ore to the stockpile and to be in a central location from the open pits. The facility will be designed to hold roughly 75,000 tonnes of ore.

18.3.1 Crushing Plant Site Layout

SGS completed the design of the crushing plant and stockpile, agglomeration, and conveying system, ADR plant and on site power generation plant and substation.

Multiple process crushing site locations were considered by the design team. The current process plant site adjacent to the leach pad pit haul road was selected as the preferred location. The crushing plant pad is oriented to work with the existing topography and existing road connections.

18.4 Process Plant Site

The ADR and power generation plant locations where selected based on the previous operation constructed pads south of the existing leach pad. The advantages of having the plants at this location include reduction in earthwork and proximity to the ponds.

The process plant site location selected for this study satisfies the study considerations and forms the basis for the material take-offs and cost estimates.

18.5 Crushing and Process Plant Site Preparation and Earthworks

The crushing plant pad is proposed to be constructed by mass-excavation, removal, and transport of exceeding material to a designated location by the project geotechnical engineer; however, a maximum of 1 kilometer has been assumed due to proximity to potential waste dumps.

It is likely bedrock cut and fill will be required for crushing plant pad, since the elevation difference to existing surface is considerable; however, detailed geotechnical information was not available in the area and geotechnical investigations are warranted to confirm and finalize the volumes based on material type and construction methods.

Top soil removal, scarification and compaction to subgrade is anticipated for the ADR plant construction, as well as for the preparation of the power plant pad.

18.5.1 Crushing and Process Plant Internal Roads

Roads adjacent to and within the plants have been provided for access service and maintenance, including the power generation plant. The construction of the roads is accounted for in the earthwork and site preparation volumes including the following:

- Rerouting of an existing road at the northerly limit of the crushing plant;
- Internal roads within the ADR plant;
- Internal roads within the power plant area; and
- Safety berms around the crushing, ADR and power plant.

Figure 18–3 shows the layout of the proposed crushing facility.

18.5.2 Tanks and Miscellaneous Equipment

Site work is also included for the pump station near Laura Creek and the power plant, as well as minor auxiliary equipment such as diesel tanks, compressors, etc.

18.6 Temporary Stockpiles for Recovered Heap Leach Material

Re-process material removed from the heap leach that is above a 0.30 gpt cut-off will be stored in one of three areas to allow for short term blending with ROM ore, or longer term storage until required for processing at the end of the mine life. A short term stockpile for recovered leach pad material will be located adjacent the ROM stockpile and will accommodate up to 150 kt of material. A second re-process stockpile will be located along the east side of the heap leach facility and will accommodate up to 150 kt of material. A third, long term stockpile for uncrushed re-process material will be located within the historical Pacific pit and will accommodate up to 1.3 Mt of uncrushed re-process material.

Conceptual stockpile pads were designed using a 2 to 1 fill slope. Fill construction material for the pads could consider use of recovered leach material that is below the 0.3 gpt cut-off.

18.7 Concrete Work

Structural reinforced concrete for the construction of the crushing plant equipment supports, process plant equipment supports, building foundations, containment walls and curbs, housekeeping slabs, tanks and miscellaneous pads have been included.

Concrete is anticipated to be mixed on site by means of a batch plant installed and operated by the general contractor, and supervised by an independent testing laboratory. Installation of reinforced concrete typically includes casting in place, formwork, reinforcing steel, inserts and additives, finishing of floor slabs and weather protection.

The concrete aggregates and cement are to be delivered to the batch plant on a scheduled basis according to casting quantities and contingency storage. Concrete pours are to be accommodated by the contractor through the use of additional rental equipment and man-power as required.

18.8 Steel

Structural steel is considered for equipment support and maintenance platforms at the ADR plant and at the crushing plant.

The capital costs in Section 21 include supply and erection of shop and field fabricated structural steel depicted in the general arrangement drawings.

Fabricated structural steel is to be delivered to the site on a scheduled basis according to the erection capacity and laydown area capacity.

18.9 Site Geotechnical

18.9.1 Foundations

Buildings will be founded on horizontal conventional spread footings or other concrete foundations, covering a layer (thickness will vary) of compacted free-draining sand and gravel above the subgrade material. Insulation and permafrost considerations will be taken into consideration for foundation construction.

18.9.2 Site Grading

The following significant site grading work will be required during the construction phase of the operations:

- Heap leach pad cell construction (Cells 8, 9 and 10);
- Truck shop area and diesel tank area;
- ROM stockpile area;
- Crushing area grading; and
- Terraces for infrastructure above the ADR facility.

Where slope stability concerns exist, it is recommended that all relevant areas are evaluated by a qualified geotechnical engineer prior to completion of construction plans. In the interim the recommended cut-slope angles for various areas required for infrastructure construction are shown in Table 18-2 below.

Table 18-2: Cut and Fill Slopes for Project Infrastructure Areas

Area	Cut slopes	Fill slopes	Cut material	Fill material	Notes
Truck shop and diesel tanks	1:1	1:2.5	Weathered rock/ overburden	Structural fill*	The truck shop will require fill on the northern side of the access road
ROM Stockpile	N/A	1:2.5	None	Waste rock from heap leach pad excavation or open pits	Fill required to level area to a gradient of less than 5%
Crushing area	Layout subject to crushing facility design				
ADR facility	No site grading required				
Terraces above ADR facility	1:2	1:2.5	Weathered rock/ rock and overburden	Structural fill*	New cuts/fills require geotechnical assessment due to steep slopes

* Structural fill as determined by a qualified geotechnical engineer

18.10 Water Consumption

The existing Water Use License provides that water to be consumed by the Brewery Creek Property from various sources as described below. A small water treatment plant for the mine camp will be constructed with a capacity to treat roughly 36,000 litres per day, with potable water either trucked or piped to the significant workplaces, such as the offices, the truck shop and the ADR facility.

18.10.1 Permissible Water Extraction Points

WUL #QZ96-007 states that water may be extracted from the following sources for the Brewery Creek operations, at a total rate not to exceed 2,824 m³ per day, and note more than rates specified for each source:

1. A maximum of 2,724 m³ per day, from Laura Creek.
2. A maximum of 50 m³ per day from groundwater well BC-23.
3. A maximum of 50 m³ per day from Lucky Creek.
4. A maximum of 50 m³ per day from Pacific Creek.
5. A maximum of 50 m³ per day, for emergency purposes, from Lee Creek.
6. A maximum of 50 m³ per day, for emergency purposes, from the North Fork of the Klondike River.

Golden Predator is required to maintain a water quality surveillance program in terms of the water licence granted.

18.10.2 Water Supply

Water is to be obtained from Laura Creek by means of submersible pumps, which will feed a gathering tank with booster pumps near the creek. The main supply line to the project is an overland pipeline with 1 metre of earth cover, which will transmit water for about 1.5 kilometers up to the plant storage tank and will branch to the campsite water storage/treatment facilities. Fire water, dust suppression, potable water and process water is to be supplied from the storage tank for the process and crushing plant as needed.

18.10.3 ADR Facility Water

The ADR facility water will be used primarily to make up water losses that occur during operations. This is not expected to be a substantial quantity of water, as water is treated and recycled within the process.

18.10.4 Truck Shop, Offices and Camp Water

Water for the office building and mine camp will be from the treatment facility and will be intended primarily for domestic use. Water for the truck shop and wash bay will be from a water source tapped prior to treatment for domestic use.

18.10.5 Fire Water

The fresh water tank will supply fire water to the automatic sprinklers, standpipe systems and yard hydrants. The water will be fed from the bottom of the freshwater/firewater tank to the fire protection system to ensure fire water availability at all times.

The tank will maintain a sufficient firewater reserve volume and sufficient residual pressure head will be provided at the ancillary buildings. Firewater storage volumes and flow rates must be verified by Golden Predator and the insurance carrier prior to final design.

18.10.6 Potable Water Treatment (Camp)

Golden Predator intends to install a potable water treatment plant for the offices and operation facilities. Commercially available units are available, including units designed for cold climates. Brewery Creek operation will require roughly 18 m³ of treatment per day. Treated water will be used for the safety eye wash and safety shower stations. The treatment will comply with the drinking water standards of Canada.

18.11 Waste Water Management

The following sources of waste water will result from mining and processing operations at Brewery Creek, other than the ADR facility:

1. Sewage and grey water from offices and operational site washroom facilities;
2. Contaminated waste water from the truck shop vehicle wash bay; and
3. Run-off from roads and operational sites such as the crushing facility.

18.11.1 Sewage and Grey Water Disposal

A report has been completed for the installation of an upgraded sewage and grey water facility for the Brewery Creek camp, June 2012 entitled “Existing On-Site Sewage Disposal System Septic Tank Replacements” (EBA, 2012). This report details the completed expansion of the sewage disposal system, which is of adequate capacity for a camp expansion for up to 120 staff, which exceeds the requirements for the purpose of the Brewery Creek operating phase.

The completed sewage disposal system consists of a septic field including two insulated trickle tanks and a septic tank.

Golden Predator will construct a lined sump at the ADR facility to contain grey water and black water from the ADR facility sanitation system and the laboratory, which will be pumped out on a regular basis for removal to Dawson City. This lined sump will not be constructed for hazardous materials such as reagents used in the plant.

18.11.2 Truck Shop Wash Bay Water

Truck shop wash bay water will be diverted to a 10 m³ three chamber oil and grease trap. The outlet of the oil and grease trap will flow into a trickle tank or the same system as sewage and grey water as described above. The oil and grease trap will have inspection hatch covers, to enable regular monitoring of sediment accumulation and removal of sediment on a regular basis. The recovered oil and grease will be placed in drums for recycling, incineration or burning in a waste oil burner.

18.11.3 Run-off Treatment

Provision has been made for construction of diversion ditches at the plant site and around the pits and waste dumps for the PEA. No design of water management structures has been undertaken and it is suggested that a system of diversion ditches or trenches be excavated to separate dirty and clean water run-off. The purpose of this system of diversion is to allow for settling of higher levels of suspended solids due to the mining operations, but not for dissolved pollutant treatment. Dissolved contaminants/pollutants are not expected to be significant and can be controlled at the source using other means such as spill containment, spill trays and concrete containment areas with sumps for working with materials with potential to cause environmental contamination.

The contact water diversion should involve a diversion ditch excavated along the northern edge of the heap leach, past the proposed truck shop location, subsequently running down the western side of the road between the camp and the ADR facility terminating north of the current waste management facility. At the incinerator a small depression exists which is proposed to be used as a retention pond to enable settlement of suspended solids. It is proposed that an unlined diversion trench is excavated towards a series of berms on the southern end of the ADR facility. This structure would be designed for average year rainfall, on the basis that abnormal flows or any event greater than a 1:25 year flood event would not be contained.

In general, contact water would be intercepted downstream of area disturbed by mining. The interceptor ditches would be constructed as ongoing water management during mining and location would depend on the mining schedule. The interceptor channel would typically be V-shaped with rock lining to prevent erosion. Where steeper gradients are required or at bends addition erosion protection may need to be provided.

Typical interceptor ditches may also include roadside swales that intercept sediment-laden water from heavily trafficked areas. Where possible runoff collected by these interceptor ditches could be routed to one of the pond facilities located at topographic low points that facilitate gravity drainage and settling.

18.12 Solid Waste Management

Golden Predator has a permitted Waste Disposal Facility on site, Permit 81-047, issued by Environment Yukon allows Golden Predator to incinerate and manage wastes on site. During the operations solid waste produced by the mine camp will be collected and stored in a waste bin prior to incineration or burial.

The incinerator will be located away from camp and will have capacity to dispose of all waste produced by camp activities. In addition to the incinerator, a landfill site in accordance with Yukon Solid Waste regulations class 2 will be permitted and constructed in accordance with Yukon guidelines for commercial landfills. Landfill sites will be constructed outside of any 1 in 25 year flood event zone and at least 1.5 km away from any drinking water well head. Landfill sites will be constructed to minimize leachate, odours, wildlife attraction and aesthetic disturbance. In addition the management and construction of the site will be undertaken in a manner which minimizes wildlife attraction.

The PEA includes an estimated cost for construction of a hazardous waste storage pad. This pad will be constructed to the west of the ADR facility, as a lined concrete pad. The facility will be used to storage empty packaging and general hazardous waste such as old oil from the operations. Hazardous waste will be kept segregated prior to off-site disposal or recycling by third parties.

The following facilities are thus envisaged for solid waste management:

- A bear proof general waste storage area at the camp and offices with bins for recyclable materials.
- An incinerator for putrescible and non-recyclable combustible waste.
- A landfill for non-recyclable waste, non-combustible waste and incinerator ash.
- A scrap yard for storage of processing plant and mobile machinery metal waste.
- A special waste storage facility for hazardous wastes including:
  - Reagent containers and packaging;
  - Old oil and grease containing waste;
  - Used filters for mining machinery; and
  - Miscellaneous hazardous waste.

18.13 Support Infrastructure

The following general support infrastructure is described for Brewery Creek operations.

18.13.1 Fuel Storage and Distribution

Fuel for the operations will be trucked to site from an existing bulk fuel storage facility and stored in two locations on site. The two locations for the fuel storage facilities are shown in Figure 18–2 and Figure 18–4. The diesel main storage facility will be located at the truck shop and will comprise four 50,000 litre tanks to ensure that there will be adequate storage for peak usage as shown in section 16, Table 16-18. In accordance with discussions with the fuel supplier, this facility would be constructed by the fuel supplier and in accordance with Yukon fuel farm regulations.

During the mining season, the tank farm will have sufficient capacity for approximately 10 days of mining operations. The fuel storage facility will have secondary containment in the form of a lined concrete foundation, with curbed sides. The concrete spill containment would need to have a volume of 110% of the largest tank on the foundation, in other words 55 m^3. A pumping and fuelling station will be included as part of this fuel storage facility.

18.13.2 Propane

Golden Predator will continue to use the current propane storage facility at the camp, with the addition of a second facility at the ADR. During Viceroy Operations this tank was on the southern edge of the solution ponds. The location of this tank is not expected to be critical and will be undertaken as determined during negotiations with the propane supplier.

18.13.3 Explosives Storage

An estimated seven tonnes of explosives will be used per day for a 7,500 tonne per day operation. Storage capacity required should be in the order of 40 tonnes of ANFO, approximately 900 boosters, 900 nonel tubes and 30 detonators. To fulfill this, the following infrastructure is required:

1. A 40 tonne ANFO silo or two storage facilities for bagged ANFO (Type 4 magazines),
2. One type 4 magazines for blasting accessories, and
3. One type 6 magazine for detonators.

The explosive storage facilities will be located at the site of the rehabilitated Blue pit, as shown in Figure 18–2. The location will be in excess of 1.2 km from any permanently inhabited area (camp). The two magazines and the ANFO storage facilities will be constructed in excess of 105 m apart as per the Explosives Regulations distance tables.

18.14 Buildings

Much of the building construction planned for the Brewery Creek Property will consist of re-establishing infrastructure that was part of previous operations. The mine camp, offices, and warehouses are to be re-established in an existing clearing. This clearing is the same location that previous camp, offices and warehouses were located at the western access road entrance to the property. A new truck shop will be constructed as per the mining contractor’s requirements. This facility will be located near the existing office building and planned operational buildings. Figure 18–2, Figure 18–4 and Figure 18–5 show the locations of the permanent mine infrastructure.

The ADR building will be located on the same location as the previous facility, with the assumption that the existing foundation will need to be removed and replaced.

18.14.1 Process Plant Building

A building for the process plant has been included as a prefabricated aluminum fame and fabric type building. The building will house the ADR system, refinery and reagent mixing facilities. A portable container with forklift access has been included outside the process plant building for storage of cyanide. Portable containers are also provided within the ADR building for Motor control centers.

18.14.2 Truck shop

The truck shop will be constructed at the location shown in Figure 18.5. The ultimate dimensions are estimated to be approximately 25 m x 50 m. This is sized to allow for four maintenance bays and storage for lubricants and equipment spares.

The truck shop will be a pre-engineered structure to reduce construction time and costs. Within the pre-engineered structure will be a concrete slab and insulated rooms for employee comfort. The site on which the truck shop will be constructed will be underlain by a synthetic liner such that pollutants from the service bays or maintenance activities are contained.

The mining contractor will service and maintain the equipment supplied within the truck shop facility. Parts will be stocked and controlled according to the needs of the mining contractor. One of the four bays will function as a welding bay and as such applicable safety installations will apply. A wash bay will also be included in the truck shop with appropriate sumps and drainage into a contained storage area. The sump water from the wash bay will be tested for hydrocarbon content and treated as required. The truck shop will have power supplied either by diesel generator or the Yukon power grid. There are currently trade-off studies being conducted to evaluate the feasibility of installing generators or connecting the mine to the power grid.

The truck shop will be in operation year round and will be insulated and heated in accordance with the temperature requirements for personnel working on equipment.

All waste oil will be used at the ADR facility boilers and any hazardous materials such as diesel filters will be stored in a special waste storage area.

18.14.3 Warehouse

A warehouse is planned to be constructed to support the Brewery Creek operations. The dimensions will be approximately 15 m x 30 m. One use for the warehouse would be the storage of parts and equipment that would support the maintenance of the mining equipment fleet.

The warehouse will be located adjacent to the truck shop and camp facilities in an area that will require minimal brush clearing and earthwork. Prior to construction, a geotechnical investigation will need to be carried out to identify complications and hazards associated with construction in the planned area.

The warehouse will be similar to the truck shop in that it will be a pre-engineered structure.

18.14.4 Mine Dry

Currently Golden Predator has a small mine dry structure adjacent to the current camp buildings. A larger dry facility will be constructed for the mining contract works, the crushing facility staff and the ADR facility staff.

18.14.5 Assay Lab

The Brewery Creek assay lab will be located on a concrete foundation east of the Viceroy ADR facility location, refer to Figure 18–3. Tetra Tech EBA has obtained costs and details for the assay lab. These structures are "design built" to the client's needs allowing for customization into an assay lab. The dimensions of the assay lab are to be 3.1 m x 12.2 m. These structures are built to withstand the harsh conditions typically encountered on a northern mine site.

18.14.6 Substation Buildings

The substation will be located adjacent to the generator location, on the terraces above the ADR facility. Refer to Figure 18–4 for the position of the substation and generators.

18.14.7 Administration Buildings

Golden Predator proposes an increase in the capacity of the administration offices for the construction and operation phases of the project. The additional offices will be placed near the current location of the offices and camp, refer to Figure 18–5.

18.14.8 Restrooms

A container building is provided in the crushing plant area and at the process plant for operations staff.

18.15 Site and Access Roads

18.15.1 Access Roads

No changes are anticipated to the access road in order to commence with proposed mining operations. The access road is in good condition, and may require surfacing and profiling from time to time. Golden Predator has equipment on site to be able to undertake this. During the construction and operation phases, there may be times when heavy equipment is brought onto site. Of concern is not the state of the access road but the two bridges that need to be crossed. Golden Predator will ensure that axial loads are within the design limitation of the bridges.

18.15.2 Mine Site and Haul Roads

18.15.3 Geotechnical and Environmental

Intrusive waste rock with the least likelihood to develop acid rock and metal leaching is proposed for haul road construction. The proposed mine waste rock fill method is similar to those used during the historical mining activities which was deemed to be successful in preventing environmental impacts.

It is not anticipated that the mine site roads will have a measureable impact on fisheries resources as the roads are located at the upper reaches of the watercourses that are periodically dry. The filling across the watercourse leading to Lucky Creek will not be considered an issue as there are numerous potential barriers to fish in the creek and no fish were captured in Lucky Creek during the environmental assessment.

18.16 Power Supply and Distribution

The power requirements for the mineral processing circuit at Brewery Creek have been estimated at 3.93 MW. In completion of the study, onsite diesel power generation has been selected as the preferred option. The other power supply options include utility power and onsite generation using LNG generators. These alternatives require further analysis in later phases of the project development. If utility power is installed at a later stage, the generators will become back-up power generation.

The generators will be purchased from the supplier as contained modular units that would be transported to site in a number of sections and assembled as necessary. The power generation equipment would be tested prior to transport to site, in an effort to reduce delays associated with troubleshooting this equipment. The modular power generation units will be connected to the various facilities by utilizing overhead power lines and buried armoured cables as required. The onsite fuel storage facility will be designed to account for the power generation facility's fuel supply requirements. The number of generators in the configuration will allow for minor and major maintenance to be carried out without placing restrictions on the amount of power available for the operation of the mine site. The preliminary generator and substation location can be seen in Figure 18–4.

The generators will be sized to supply power for the entire project requirement of 3.93 MW, which includes a 19 percent contingency for future additional loads during detail engineering. The Brewery Creek Mine substation includes diesel power generation and main switchgear for distributing power on site, and also space for a utility transformer yard. This substation will be located near the process plant. The diesel power generation will be in an enclosed building that houses the diesel generators, tanks and an electrical room for the main switchgear and generator control.

The diesel generator building will house four 1.75 MW generators rated 4.16 kV, continuous duty, with three operating and one spare. Power will be distributed on site via the main switchgear and overhead power lines, rated 4.16 kV, to the crushing & agglomeration area, process plant area, the fresh water pump station, the truck shop and office & camp area.

18.16.1 Switchgear and Motor Control Centers

Electric power generated and distributed on site is rated 4.16 kV, 3 phase, 60 Hz. The main switchgear includes power circuit breakers for receiving power from each generator and feeder circuit breakers for distributing power on site, rated 4.16 kV, via overhead power lines. The main switchgear will also include space for a future connection to the utility transformer. A stand-alone digital master control system will control the use of the generators based on the load demand. If utility power comes online, synchronization will be needed for the generators to provide backup power for supporting all power requirements. The power distribution on site is rated 4.16 kV for medium voltage applications, 480V for low voltage applications and 120 V for controls and instrumentation.

Power will be distributed on site via a radial network with two overhead power lines. One power line will feed the crushing & agglomeration area, truck shop and office & camp area, and the second power line to feed the process plant & fresh water pump station.

Power for primary crushing will be supplied via a 4160 - 480 V, 750 kVA transformer that feeds a 480V motor control center (MCC). Power for secondary & tertiary crushing will be supplied via a 4160V switchgear feeding two crusher motors (500 horsepower, each) and a 4160 - 480 V, 750 kVA transformer that feeds a 480V MCC. Power for agglomeration will be supplied via a 4160 - 480 V, 500 kVA transformer that feeds a 480V MCC. All motor control centers supply 480V power for all crushing & agglomeration low voltage applications.

Power to the process plant will be supplied via a 4,160V switchgear feeding two medium voltage motors (500 horsepower, each) and a 4160 - 480 V, 2000 kVA transformer that feeds 480V Motor Control Centers, which supply 480V power for all low voltage process applications.

Power to the fresh water pump station, truck shop and office & camp locations will be supplied via 4160 – 480 V transformers at each respective location and distribute power to all equipment using distribution panels or service entrance panels or individual motor control panels.

The electrical design, for the study, is based on the requirements of the latest edition of the National Electrical Code (NEC) and the detailed design phase will comply with federal, provincial and local regulations and standards. All equipment selection will be based on NEMA and ANSI design. The details for all electrical installations are reserved for the detail engineering phase. Refer to all single line diagrams which describe preliminary plans showing 69 kV power supply to site, and 4.16 kV power distribution on site for 4.16 kV medium voltage applications and stepped down to 480 V for low voltage applications.

18.16.2 Instrumentation

The instrumentation and controls design, for the study, is estimated based on 'Supervisory Control and Data Acquisition System (SCADA)', a computer based data gathering system that will control and monitor all process operations including cyanide detection. The SCADA system will use remote termination devices to channel appropriate control and monitoring signals from field locations back to the central processing unit (CPU) computer where an operator can physically operate equipment from his computer work station. The configuration of the SCADA will be based on the latest industrial standards. A programmable logic controller (PLC) system will be installed in respective areas, gathering information from the input and output signals from instruments and motor control equipment. The majority of PLCs, I/O racks, MCCs, VFDs, and other control devices will be located in air conditioned electrical rooms. The SCADA will process and record all communications with respective PLCs. An uninterruptable power supply (UPS) will provide power to each PLC. The details for instrumentation and control are reserved for the detail engineering phase.

18.16.3 Site Wide Distribution

Electric power requirement for the Brewery Creek Mine Project is described in Table 18-3.

Table 18-3: Power Distribution for Brewery Creek Mine Project

Area Description	Power (MW)
Crushing + Agglomeration + Shop/Offices/Camp	
▪ Crushing	1.36
▪ Agglomeration	0.33
▪ Shop	0.15
▪ Offices/Camp	0.11
Sub-Total	1.95
Process Plant + Fresh Water Pump Station	
▪ Process Plant	1.25
▪ Fresh Water Pump Station	0.10
Sub-Total	1.35
TOTAL (Brewery Creek Mine Project)	**3.30**
Contingency (19%):	0.63
TOTAL with Contingency (Brewery Creek Mine Project)	**3.93**

The power demand for the project is about 3.3 MW and with a 19 percent contingency for future additions in detail engineering, the total power demand is about 3.93 MW. The future additions are based on contingencies for certified vendor information and additional equipment added during detail engineering. The power requirements were established using the process design criteria, equipment list, and electrical load study by SGS Metcon (SGS Drawing 00-E-001, 2014).

18.16.4 Grid Power Supply Trade-Off

For completion of the PEA, the alternative of using grid power for providing power to the proposed mining operation at Brewery Creek was examined. The details are included here for future reference.

Golden Predator commissioned an independent conceptual study to estimate the cost of connecting the Brewery Creek Mine to the Yukon Energy Corporation (YEC) network via the existing L177 69 kV line. Four options were considered for connecting the mine site to the Yukon power grid with the two most attractive being presented in a report produced by BBA Engineering (BBA, 2012).

The most attractive options for connecting to the Yukon power grid consist of two different routes of 16 km and 27 km and two different voltages of 69 kV and 34.5 kV. The evaluation of the two different routes included consideration of power system impacts, transmission line routing, tap point and Brewery Creek Mine substations, step-down transformers, and circuit breaker requirements.

The conceptual level cost estimates of the two most attractive alternatives, including both direct and indirect costs of +/- 30%, are included in Table 18-4.

Table 18-4: Grid Power Options and Estimated Costs (source BBA Engineering)

Option	Approach	Total cost
Alternative 1	(16km – 69 kV)	$15,783,034
Alternative 2	(27km – 69 kV)	$19,073,008
Alternative 3	(16km – 34.5 kV)	$15,731,621
Alternative 4	(27km – 34.5 kV)	$17,290,497

The conceptual study included an evaluation of the cost and feasibility of the following aspects of connecting the mine to the Yukon power grid:

1. The Tap Point Substation;
2. The Transmission/Distribution Line; and
3. The Brewery Creek Mine Substation.

18.16.4.1 Tap Point Substation

The only option for the Brewery Creek Mine to connect to YEC is via the 69 kV transmission line along the Klondike Highway 2. The connection point would be located approximately 50 km southeast of Dawson City, either at the Dempster Corner, or slightly south of Dempster Corner.

18.16.4.2 Transmission/Distribution Line

The study completed by BBA focused on evaluating two possible routes for the transmission line to follow. Route A and B can be described as follows:

- Route A is the shortest at 16 km, from Klondike Highway 2 cross-country to North Fork Road.
- Route B is the longest at 27 km, along the existing Dempster Highway 5 and North Fork Road.

Routes A and B were evaluated for two different voltage levels, 69 kV and 34.5 kV, creating four alternatives. 69 kV is usually the minimum voltage for the term “transmission line”. Lines at lower voltages are typically classified as “distribution lines”.

Compared to the 69 kV transmission line, the 34.5 kV distribution line would reduce the available short-circuit level to a point where it might be difficult to start/operate some of the mine’s crusher and other larger motors. Taking into consideration that the 69 kV transmission line would be able to transfer more power, and that the 34.5 distribution line would ease the connection of new customers, further discussion is required between YEC, Golden Predator before the line with the most suitable voltage can be selected.

18.16.4.3 Brewery Creek Mine Substation

The Brewery Creek Mine Substation will require a step down transformer regardless of the transmission/distribution line voltage. The transformer will step down the voltage to 4.16 kV. For the purpose of the purpose of the study completed by BBA, 4.16k V switchgear was assumed with 1 main and 5 feeder breakers complete with protection and control. This substation configuration will be subject to change as a more accurate load list becomes available.

18.16.5 LNG Power Supply Study

A conceptual study on the viability for on-site LNG power supply as alternate to diesel power generation was undertaken by Tetra Tech EBA (2014). The study outlined the successful application of LNG fuel as power supply in the Canadian northern climates, considered potential supply options and conceptual onsite facility layouts, and review regulatory requirements for construction and operation of LNG power generating facility.

Further design and cost estimation would be required to fully evaluate the use of LNG as a viable fuel option. LNG fuel was not considered in the economics or as the base case for this study.

18.17 Lighting and Grounding

The capital cost estimate, for the study, includes lighting estimates based on lighting design per Illumination Engineering Society (IES) published guidelines. Lighting will include high pressure sodium type fixtures for exterior use, metal halide lighting fixtures for indoor low bay applications and fluorescent lighting fixtures for offices, electrical rooms, etc. All electrical installations including wiring, tray, connections, grounding, lighting will be installed in accordance with the NEC and adhere to the federal, state and local code. The cost estimation for the study includes estimates for grounding based on a continuous above-ground system of grounding conductors and bond wires that will interconnect all equipment (electrical, mechanical and structural) to a single buried ground grid system for safety. A test well will be provided for periodically testing the resistance of the grounding system. The details for lighting and grounding design are reserved for the detailed engineering phase.

18.18 Communication

Golden Predator currently has a communication system in place, which will not require upgrading other than purchase of additional handheld units for distribution to operation staff. The system consists of radios with fixed stations at the camp and offices, vehicle mounted units and hand held radios. External communications are provided by satellite link-up providing both voice and data communications.

18.19 Fire, Life and Emergency Response

Golden Predator currently has emergency response equipment in place including a first aid room, an ambulance, firefighting equipment and a communication system with emergency protocols. The following fire emergency response facilities will be provided for the operational phase of the project:

1. Additional first aid equipment at the ADR facility and the truck shop including:
    a. First aid boxes level 1 and level 3;
    b. Stretchers;
    c. Blankets; and
    d. Spill response equipment.
2. A fire suppression/hydrant system for the truck shop and ADR facility;
3. Additional fire extinguishers of the appropriate classes for the camp and various operational areas;
4. Muster stations at the administration offices, the truck shop and the ADR facility; and
5. An emergency response operations control centre (an operational office set-up with appropriate communications equipment for available for use during emergencies).

19.0 MARKET STUDIES AND CONTRACTS

19.1 Gold Marketing

Gold production from the Brewery Creek is likely to be sold on the spot market through marketing experts or agencies retained by Golden Predator. The large numbers of available gold purchasers, both domestically and internationally, allow for gold production to be sold on a regular and predictable basis, and on a competitive basis with respect to the spot price.

Golden Predator expects that terms contained within any sales contract entered into would be typical of, and consistent with, standard industry practices.

19.2 Equipment Leasing Contracts

The Project contemplates the leasing of major mining equipment. This is common practice in the industry as it reduces initial capital expenditures as well as assisting the company in maintenance of the equipment as leased equipment is also often subject to MARC (maintenance) agreements with the manufacturer. All manufacturers offer leasing option for their equipment, so there is very little risk in the company's ability to lease equipment.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Historical Studies

Numerous environment and social impact studies were completed on the Property since the early 1990s. Many of the baseline studies on the site prior to Viceroy operations were conducted as part of the Initial Environmental Evaluation, under the CEAA (Canadian Environmental Assessment Act) environmental assessment process. These studies included baseline fisheries studies by Norecol Environmental Consultant in 1990, a heritage resource study conducted by Fedirchuk McCullough & Associates in 1990, and a soil survey conducted by Norecol Dames and Moore in 1991. In addition, the following environmental studies have historically been conducted on site:

- Fish Habitat and Utilization Assessments Brewery Creek Mine Site Vicinity 1997-1998, Lower Laura, Lee and Pacific Creeks January, 1999 Prepared by: White Mountain Environmental Consulting,

- Results Summary of Fish Tissue Analysis – 2001 *Brewery Creek Mine* Prepared for: Viceroy Minerals Corp by Access Consulting Group,
- Biological Monitoring Program at Brewery Creek, Y.T. 2005, prepared by LaBarge Environmental Services,
- Results Summary of Fish Tissue Analysis – 2001, prepared by Access Consulting Group,
- Brewery Creek Mine Hydrological Investigation of Lower Laura Creek, 1998 prepared by Access Consulting.

Further, regular sediment and benthic sampling programs have been carried out on site, as well as hydrology, hydrogeology and water quality sampling/monitoring programs. There is approximately twenty years of environmental baseline data for the Brewery Creek Property that is housed in the various Water Use Licence submissions, for both its production and reclamation phases.

Tetra Tech EBA is aware of the documentation listed above, but has not conducted a detailed review of the contents of these reports.

20.2 Recent Studies

20.2.1 YESAB Submission

On January 16, 2013, Golden Predator submitted a project proposal for the re-activation of the Brewery Creek mine to the Designated Office in Dawson, Yukon. On February 14, 2013, the Designated Office in Dawson informed Golden Predator that the project would have to be reviewed by the Executive Committee and that Golden Predator should re-submit the project proposal to that committee for review. Work is underway to prepare the Project Proposal, however, a document had not been re-submitted to date.

20.2.1.1 Geochemical Characterization of Waste Rock

Geochemical characterization and waste rock management was addressed in this submission. The Waste Rock Management Plan (WRMP) for the Brewery Creek mine was originally submitted to the Yukon Water Board for water licence QZ94-003 in December 1996. The WRMP is structured framework, which guides the waste rock management process from pre-development characterization through production tracking and post production inventory and monitoring. The WRMP for the reactivation project is currently being revised based on additional geochemical characterization work that has been completed at the site but the basic structure and approach presented in it has been found to be valid based on operations experience from the first period of mining.

The pre-mine studies indicated generally low potential for net acid generation and metal release (fully developed ARD) across the site. Some samples tested indicated potential for metal release without net acidity. The following were identified as predominant mechanisms for metal release from Brewery Creek Mine waste rock:

- Short term metal release as a result of dissolution of existing secondary weathering products (referred in this section as stored products), and
- Short and long term metal release due to oxidation of sulfide minerals, with acidity buffered by neutralizing minerals.

Predominant release mechanisms by rock type identified in the pre-mining study are summarized in Table 20-1 below.

Table 20-1: Release Mechanisms by Rock Type

Rock Type	Short and Long Term (dissolution and oxidation)	Short Term Only (dissolution)	Long Term Only (with local net acid generation)
Quartz monzonite		X	
Altered quartz monzonite	X [1]		
Limonitic altered quartz monzonite	X		
Argillite		X	
Graphitic Argillite	X		O [2]
Shale		X	
Greywacke/chert-pebble conglomerate	X		O [2]

[1] Dissolution observed in AQM samples may be a function of time since collection.

[2] A small portion of samples from this unit indicate net acid generating conditions. However, the overall balance is acid consuming.

20.2.2 Site Hydrology and Water Balance

Hydrological data collected between the years 2006-2011 was used to calculate monthly average discharge measurements at seven hydrometric stations on the Brewery Creek Mine site. These stations listed in the following table:

Table 20-2: Brewery Creek Hydrometric Stations

WQ / Hydro Stn.	Description	Historic Hydro. Stn. ID	Monitoring Type	Catchment Area (km²)	Min. Elev. (m)	Max. Elev. (m)	Average Slope (%)
BC-01	Laura Creek above ditch road (below mine activities)	H2	continuous	27.1	527	1187	8.7
BC-02	Carolyn Creek above Laura Creek (below heap)	H4	continuous	3.0	606	856	12.2
BC-03	Laura Creek above Carolyn Creek	no stn.	continuous	16.2	613	1184	-
BC-05	Pacific Creek above Lee Creek (below Moosehead and Big Rocks)	no stn.	continuous	31.5	549	1180	6.4
BC-33	Lee Creek Above Pacific Creek (below Moosehead)	no stn.	continuous	187.7	546	1859	-
BC-34	Lee Creek at Mine Access Road Bridge	H3	manual	225	520	1680	3.9
BC-35	Upper Pacific Creek South Arm (above Big Rocks and Moosehead)	no stn.	continuous	8.2	638	1180	-
BC-35R	Upper Pacific Creek North Arm	no stn.	manual	6.4	6.38	1158	-

Table 20-2: Brewery Creek Hydrometric Stations

WQ / Hydro Stn.	Description	Historic Hydro. Stn. ID	Monitoring Type	Catchment Area (km²)	Min. Elev. (m)	Max. Elev. (m)	Average Slope (%)
	(Reference) (above Moosehead and Big Rocks)						

Average monthly flows at all stations were highest in May (BC-1: 0.251 m³/s; BC-2: 0.339 m³/s; BC-3: 0.296 m³/s; BC-5: 0.473 m³/s; high flows for the rest of the stations require more data and collection is on-going to increase data set) , and annual low flows typically occur in March. Predicted mean monthly run-off was also calculated for each site and data is located in the table below:

Table 20-3: Predicted Mean Monthly Runoff During an Average Precipitation Year

Month	Discharge (m³/s)					
	BC-1	BC-3	BC-5	BC-33	BC-34	BC-39
Jan	0.036	0.021	0.041	0.291	0.349	0.048
Feb	0.031	0.018	0.036	0.249	0.298	0.042
Mar	0.031	0.018	0.036	0.242	0.290	0.042
Apr	0.054	0.032	0.062	0.395	0.474	0.073
May	0.686	0.410	0.797	3.466	4.155	0.934
Jun	0.562	0.336	0.654	3.737	4.479	0.766
Jul	0.219	0.131	0.255	1.930	2.313	0.299
Aug	0.219	0.131	0.254	1.746	2.093	0.298
Sep	0.187	0.112	0.218	1.561	1.872	0.255
Oct	0.100	0.060	0.117	0.940	1.127	0.137
Nov	0.061	0.036	0.071	0.541	0.649	0.083
Dec	0.046	0.028	0.054	0.368	0.441	0.063

20.2.3 Permits Held by Golden Predator

In September 2012, Golden Predator acquired all of the mine assets, obligations, and liabilities for the Brewery Creek Property. The site is authorized under a suite of project approvals, including:

- Quartz Mining Licence (QML) A99-001 expires 2021 (mine production and closure, monitoring and inspections, etc.)
- Type A Water Use Licence (WUL) QZ96-007 expires 2021 (mine production and closure, mine and camp water use and waste deposit, water management and monitoring, etc.)
- Type B WUL MN12-038 expires 2022 (camp expansion, camp water use, disposal of domestic waste).
- Class IV Mining Land Use Approval LQ00364 expired 2022 (120 person camp, advanced exploration activities)
- Waste Management Permit #81-047 (camp domestic waste facility and incinerator, special waste storage and air emission permit)

The Project will require an amendment of the existing mine production WUL QZ96-007 and QML A99-001.

20.2.4 Ongoing Environmental Studies

The following ongoing monitoring and studies are being completed on the property by Golden Predator personnel and various independent and consulting organizations:

- Hydrogeological Baseline Assessment;
- Wildlife Assessment;
- Benthic Invertebrate Study;
- Heritage Resource Assessment;
- Socio-economic Impact Assessment;
- On-going baseline and QZ96-007 Water Use Licence compliance water quality and hydrology sampling and monitoring program; and
- Fisheries Impact Study.

20.3 Tr'ondëk Hwëch'in First Nation Socio Economic Accord

Golden Predator and the Tr'ondëk Hwëch'in First Nation entered into an Amended and Restated Economic Accord for the Brewery Creek Property which updated the existing Socio Economic Accord that was in effect since its inaugural signing with Viceroy in 1996.

Key aspects of the Socio Economic Accord include:

- THFN support for the Project;
- THFN endorsement for the Company's permitting applications, with a clear process for THFN to review and provide input prior to filing, and a mechanism to expeditiously address and resolve any concerns THFN may have;
- A consistent and clear process for communication on all matters pertaining to the Brewery Creek Project and resolving any disputes that may arise;
- Preferential employment and economic development opportunities for TH businesses and citizens;
- THFN acquiring an equity interest in the Company, and participating in profit sharing from operations beyond the original mine plan;
- Funding for training and scholarships for TH citizens; and
- An annual grant to a community legacy project for the broader community of Dawson.

21.0 CAPITAL AND OPERATING COSTS

This section describes the methodology used to prepare the capital and operating cost estimates. The total pre-production capital cost estimate including contingency, indirect and owners costs cost is US$89 million and the life of mine average operating cost is US$19.95 per tonne processed and US$778 per ounce saleable.

These costs are based on the project production rates and design criteria established by SGS - E&S Engineering Solutions Inc. for the proposed operation. It is important to note that the capital cost estimate does not include sunk costs such as drilling, metallurgical test work, or prior studies undertaken to date.

The sources of information for the cost estimates include:

- Quotes and budgetary estimates by suppliers and contractors;
- Information gathered during site visits;
- Topographical surveys of the mining area;
- Historical cost and operational parameters from the Viceroy operations;
- Current cost information relevant to Golden Predator's operations in Yukon;
- Local knowledge regarding costs and prices for the Yukon;
- Similar mine financial information and feasibility assessments; and
- Industry cost information such as the Costmine, published by Infomine USA, Inc.

Additionally, the following assumptions are applicable to the capital and operating cost estimates:

- A maximum of 230 operating days per annum for mining, crushing and heap leach loading;
- Daily production rate of 7,500 tonnes per day;
- Annual production rate of 1,725,000 tonnes per annum;
- Ore density of 2.57 tonnes/m^3;
- Swell factor of 1.3;
- 24 hours per day operations;
- Two shifts per day;
- Exchange rate of US$ to CAD$ of 1:1.09; and
- Costs are reported in US$, unless stated otherwise.

21.1 Capital Costs

Capital costs have been estimated for the open pit mining, ore handling, heap leach and processing, water management, onsite infrastructure and indirect costs. Table 21-1 summarizes the capital cost estimate completed for the Brewery Creek property. General and administrative costs for years -2 and -1 are included as owner's costs during the construction period.

Table 21-1: Summary of Capital Costs

Capital costs in US$000						
Capital cost item	**Estimated initial capital**	**Contingency $**	**Contingency %**	**Total Initial**	**Sustaining including contingency**	**Total capital**
Direct						
General site	$64	$3	5%	$67		$67
Site infrastructure	$2,857	$429	15%	$3,286		$3,286
Preproduction and haul roads	$890	$	0%	$890		$890
Mining equipment	$65	$16	25%	$81		$81
Mining infrastructure	$615	$92	15%	$708		$708
Total mining and site infrastructure	**$4,491**	**$540**	**12%**	**$5,031**	**$**	**$5,031**
Processing excluding heap leach construction						
Crushing	$10,902	$2,180	20%	$13,082		$13,082
Agglomeration	$3,349	$670	20%	$4,019		$4,019
Ore stacking	$728	$146	20%	$874		$874
ADR facility and heap leach equipment	$9,918	$1,984	20%	$11,901	$4,128	$16,029
Process infrastructure	$8,159	$1,632	20%	$9,790		$9,790
Total processing	**$33,055**	**$6,611**	**20%**	**$39,666**	**$4,128**	**$43,795**
Heap leach and water management						
Heap leach including ponds	$12,764	$1,915	15%	$14,679		$14,679
Water management	$83	$21	25%	$103		$103
Total HLF and water management	$12,847	$1,935	15%	$14,782	$	$14,782
Total direct	**$50,393**	**$9,087**	**18%**	**$59,480**	**$4,128**	**$63,608**
Indirect						
Capitalised mining	$11,365			$11,365		$11,365
Process indirects	$5,550	$1,110	20%	$6,661	$33	$6,694
Mining and other indirects	$1,517	$228	15%	$1,745		$1,745
Owners costs (G & A year -2 and -1)	$5,998	$	0%	$5,998		$5,998
Total indirect	**$24,431**	**$1,338**	**5%**	**$25,769**	**$33**	**$25,802**
Total capital in US$000	**$74,824**	**$10,425**	**14%**	**$85,249**	**$4,161**	**$89,410**

21.1.1 Site Infrastructure

Tetra Tech has included $67 thousand for security and yard lights, and $3.3 million for site infrastructure including offices and camp buildings, which will be purchased as prefabricated units, including construction of foundations and installation of services. Other facilities such as workshops and storage facilities have been included in estimates for processing and mining capital.

21.1.2 Open Pit Mining Capital Estimate

Note that the base case used for the financial modelling for this PEA considers leasing of mining equipment, due to the limited mine life. As such, the estimated project capital cost does not include mining equipment. However, in order to estimate potential leasing rates, the mining fleet has been estimated as well as the purchase cost of the mining fleet. Leasing rates have been estimated to include 18% interest fees.

Mining equipment capital estimates include development capital such as pre-stripping and haul road construction as well as purchases for mining equipment and maintenance facilities. The estimate has been based on budgetary estimates, quotes, industry cost data and factoring of costs based on equipment sizing. The mining fleet is based on the mining schedule, estimated equipment productivity and time available to achieve schedule objectives. Based on the schedule for mining of pits and the old heap leach, 71% of the process feed will come from the pits. An average of 5,500 tonnes per day will be mined from the pits over the life of mine. Mining equipment is based on achieving peak production of 7,500 tonnes per day from the pits, which occurs in year 3.

21.1.2.1 Pre-stripping and haul road construction

For the PEA, the preproduction haul road construction and pre-stripping has been included in the capital cost, as open pit development costs. These costs are broken down in Table 21-2.

Table 21-2: Preproduction Pit Development and Haul Road Cost Estimate

Item	Contingency [1]	Total cost
Site haul roads preproduction equipment usage cost		**$ 294,610**
Site haul roads preproduction Culverts	48,074	**$ 368,564**
Dump and pit diversion trenches	5,513	**$ 42,263**
light vehicle access	24,069	**$ 184,530**
Pre-stripping year -2 [2]		**$ 239**
Pre-stripping year -1 [2]		**$ 11,131**

[1] No contingency is applied to costs produced through Runge Xeras™ to be consistent with approach taken for operating cost estimate

[2] Pre-stripping costs have been incorporated as capitalized mining costs in the indirect capital

21.1.2.2 Mining equipment

Note that mining equipment costs are not included in the base case capital costs as mining equipment leasing has been considered. An estimate of the mining equipment costs has been included as a basis for estimated leasing rates and for use as a trade-off study considering the option of purchase rather than lease of mining equipment.

Mining equipment sizing and numbers are based on outputs of the Runge XERAS™ which evaluates hours available and hours required. The equipment sizing is based on productivity estimates required to achieve the daily tonnage. Freight is calculated based on equipment weight and road transport rates for round trip from southern Canada to the mine site.

Table 21-3: Capital Cost Estimate for Mining Equipment

Item	Number	Freight	Contingency	Total estimated cost in USD including contingency and freight
Survey equipment	1		6,250	**$ 31,250**
Software	1		10,000	**$ 50,000**
Mining engineering equipment total				**$81,250**
Shovels 6.4 m³ bucket	1	70,574	126,081	**$ 1,386,896**
FEL 6.9 m³ bucket	1	47,400	85,140	**$ 936,540**

Table 21-3: Capital Cost Estimate for Mining Equipment

Item	Number	Freight	Contingency	Total estimated cost in USD including contingency and freight
FEL 6.3 m³ bucket year 1	1	47,400	85,140	**$ 936,540**
65 Tonne waste truck	5	52,200	441,620	**$ 4,857,820**
80 Tonne truck (59 m³ body)	3	60,000	307,400	**$ 3,381,400**
40 Tonne truck	2	39,000	133,900	**$ 1,472,900**
Backhoe excavator	1	15,000	26,500	**$ 291,500**
Dozer 13.7 m³	1	44,319	79,698	**$ 876,674**
Blast hole drill	1	47,100	84,610	**$ 930,710**
Water Truck	1	38,913	69,586	**$ 765,449**
Grader	1	22,800	40,364	**$ 444,004**
Fuel and lube truck	1	4,800	8,480	**$ 93,280**
Medium truck	5	4,200	7,420	**$ 81,620**
Light vehicles	3	1,500	12,650	**$ 139,150**
Pump	1	1,500	3,490	**$ 38,390**
Lighting sets	1	300	1,530	**$ 16,830**
Low boy truck	1	9,000	15,900	**$ 174,900**
Tire handler	1	6,000	10,656	**$ 117,216**
Compactor	1	12,000	21,256	**$ 233,816**
Welding truck	1	7,200	12,720	**$ 139,920**
Explosive truck	1	3,900	6,946	**$ 76,406**
Mobile crusher	1	22,500	39,806	**$ 437,866**
Total mining mobile equipment costs[1]				**$17,829,827**

[1] Mining mobile equipment costs have not been included in capital costs as the PEA considers equipment leasing at a rate of 18%, resulting in a cost of $3.5 million per year over 9 years for leasing, which is equivalent of $0.59 per tonne mined over life of mine

21.1.2.3 Mining infrastructure

An estimate of the capital cost of mining infrastructure to support the mining operations is provided in Table 21-4.

Table 21-4: Estimated Capital for Mine Site Infrastructure

Item	Freight	Contingency	Total including contingency and freight
Truck Shop [1]	8,000	78,035	**$ 598,270**
Truck Shop earthworks		1,995	**$ 15,295**
Tool store	2,450	1,298	**$ 9,948**
Tool shop	2,450	1,298	**$ 9,948**
Mine office	2,450	1,298	**$ 9,948**
Mine dry	3,000	8,370	**$ 64,170**

[1] Includes truck shop infrastructure such as cranes, tool store and container offices

21.1.3 Site Work (Earthworks) Costs Included in the Capital Cost Estimate

Site work costs include excavations and back fill calculations, using preliminary grading plans prepared for the crushing plant options, and rehabilitation of existing graded pads at the process plant and power generation plant.

Site work for the process plant excludes cuts and fills to provide for laydown areas, parking, office facility, and costs for utility corridor and access roads to the crushing and process plant sites, as well as upgrading access roads and additional facilities. However, these are included under the general site infrastructure capital.

The rates in Table 21-5 are applied to site work (excluding mining and haul roads, which are a different rate) for estimating the capital cost of earthworks undertaken during construction of the mine infrastructure.

Table 21-5: Earthworks Rates Used for the Capital Cost Estimate

Type of Site Work	Units	$ Rate per Unit
Mass excavation - Common Soil	m^3	$ 3.19
Permafrost excavation	m^3	$ 5.25
Structural excavation	m^3	$ 5.25
Structural excavation - rock	m^3	$ 5.93
Engineered backfill - including compaction	m^3	$ 9.48
Brush clearing	m^2	$ 0.76
General backfill / re-handle	m^3	$ 3.23
Grading cuts	m^3	$ 3.80
Subgrade preparation	m^3	$ 10.20

21.1.4 Haul Road Cost Estimates

The cost estimate assumes that the existing haul roads can be re-established for the haul operations and no significant cutting into the existing slopes or filling add-ons to side slopes are required. The estimate accounted for removal cost of organic soils and vegetation, grading and blading of the road to the haul road width, and placement of travel surface gravels. Some locations require waste rock fill including a section of haul road realignment at the Heap Leach, at watercourse locations to build up the road for proper sag vertical curve, and at horizontal curve improvement locations. Cost for supply of waste rock, haul and dump at the haul road locations were accounted for in the mining cost. The only cost included in the haul road estimate is blading and grading the waste rock fill prior to placement of surfacing gravel.

The cost estimate for surfacing gravel assumes that a mobile crusher will be used to crush mine waste rock and that the haul distance will be less than 4 km. Surfacing gravel costs are included for existing and new haul roads, site access roads for light vehicles and for the access road to Laura Creek pump station.

21.1.5 Concrete Construction Rates Used in the Capital Cost Estimate

Concrete volumes are included based on material take-offs from the general arrangement drawings. Installation costs include cast-in-place concrete, formwork, reinforcing steel, inserts and additives, float finish of floor slabs, curing materials and weather protection. The concrete cost includes aggregate, sand and cement to be delivered on-site from an outside source, including the installation of a temporary concrete batch plant on-site.

Tetra Tech EBA and SGS Metcon have estimated that concrete will cost $700 per cubic metre based on project experience in the Yukon and the recent project in Dawson City during 2012, where concrete costs were roughly $350 per cubic metre. These rates include rebar but exclude labour.

Labour is added to the concrete costs between 8 and 10 hours per cubic metre, depending on complexity of the concrete pour.

21.1.6 Construction Labour Used in the Capital Cost Estimate

Direct labour costs for the project include only those costs incurred directly in carrying out the scope of this project. Budgetary labour rates are used to develop a composite hourly construction labour rate. A summary is shown in Table 21-6.

Table 21-6: Direct Labour Costs

Disciplines	Average Pay per Hour CAD$	Crew Size	Total Cost per Hour CAD$
Project Manager	$ 110.00	0.5	$ 55
Construction Manager	$ 105.00	1	$ 105
Craft Superintendent	$ 100.21	1	$ 100.21
General Foreman	$ 94.37	1	$ 94.37
Craft Foreman	$ 80.12	3	$ 240.36
Journeyman (electrician, ironwork, millwright)	$ 78.18	6	$ 469.08
Journeyman Other	$ 72.37	12	$ 868.44
Helper	$ 60.23	12	$ 722.76
Warehouseman	$ 63.12	1	$ 63.12
Labourer	$ 63.12	12	$ 757.44
Truck Driver	$ 63.12	2	$ 126.24
		51.5	$ 3,602

The resulting rate consider a seven day - 77 hour work week paid at regular rate for all staff. Based on the crew size, a composite hourly rate is obtained by dividing the total crew cost over the number of crew members. The resulting rate is US$70.00/hour. Composite crew rates are based on the following assumptions:

- Unemployment insurance, general liability and insurance, and fringe benefits.
- Small tools, consumables and personal protective equipment.
- Site supervision and administration, indirect craft labour, mobilization and de-mobilization, and temporary site facilities.

The total construction labour hours are estimated to be roughly 100,000 hours. This averages to 160 days per crew member undertaking construction work over 13 months. A crew member will on average, be actively working for half of the project construction phase.

21.1.7 Structural Steel

Structural steel is included based on material take-offs from the general arrangement drawings. The unit pricing includes shop and field fabrication, and installation. Table 21-7 shows the unit rates and labour multipliers used to estimate the structural steel construction costs.

Table 21-7: Unit Rates and Labour Multipliers for the Structural Steel Construction Costs

Type of Structure	Construction Labour Hours per kg	Material / Steel Cost per kg
Conveyor belts	0.044 hours	$4.41/kg
Columns and beams	0.044 hours	$4.41/kg
Handrail	0.055 hours	$7.94/kg
Platform	0.051 hours	$2.54/kg
Stairs	0.044 hours	$3.31/kg
Bracing	0.265 hours	$3.31/kg
Miscellaneous (plates, bolts etc.) 5% of sub-total	0.055 hours	$4.41/kg

21.1.8 Water Management

Tetra Tech EBA has included a capital cost of $103 thousand for potable water treatment and sewage treatment. Note that the camp already has a sewage treatment system adequate for an operations camp catering for 80 people.

21.1.8.1 Fresh Water

The cost estimate for the fresh water supply system, for the Brewery Creek Property, will provide storage tank and associated booster pump station, electrical power line and piping from Laura Creek. The water system excludes the cost of any well development.

21.1.8.2 Process Piping Costs

Process plant piping is factored based on similar projects in the SGS Metcon capital cost estimate.

21.1.9 Power Supply

Power supply costs are included under the process plant capital. Electrical power supply costs use diesel powered generators as the sole source of power. The diesel generators were sized to supply power requirement of 3.93 MW.

21.1.9.1 Electrical – Power Distribution and Instrumentation

Electrical and Instrumentation costs for individual process/facility include transformation and service, wiring, cable tray, lighting and grounding within the respective areas of crushing and agglomeration area, process plant area, the fresh water pump station, the truck shop and office and camp area. These costs are based on historical data and recent project information by SGS Metcon. The power distribution on-site is rated 4.16 KV for medium voltage applications, 480 V for low voltage applications and 120 V for controls and instrumentation. The details for power distribution and instrumentation are reserved for the detail engineering phase. SGS Metcon has estimated that 4 km of overhead lines are required to distribute power through the site. The mining areas will not be provided with line power, as all equipment will be diesel or diesel generator driven.

The cost estimation for distributing power to process equipment includes the following electrical and control items:

21.1.9.2 Transformers Sizes

The following transformers have been selected based on the required power requirements at the various facilities.

- Primary Crushing: 750 kVA (4160 - 480V)

- Secondary and Tertiary Crushing: 750 kVA (4160 - 480V)
- Agglomeration: 750 kVA (4160-480V)
- Process Plant: 2000 kVA (4160-480V)
- Offices and Camp, Truck Shop: 150 kVA (4160-480V), per area
- Fresh Water Pump Station: 300 kVA (4160-480V).

21.1.9.3 Switchgear and Control

The following major switch gear will be required at the Brewery Creek Property.

- Secondary and Tertiary Crushing: 600 Amp (4.16 kV), Crusher motor starters (4.16 kV)
- Process Plant: 600 Amp (4.16 kV), Barren solution pump motor starters (4.16 kV)
- 480V Motor Control Centers (MCC) and feeder installations, lighting and grounding
- Supervisory Control and Data Acquisition (SCADA) and Programmable Logic Controller (PLC) systems

21.1.10 Processing Plant Capital

A summary of the crushing, process plant and associated infrastructure capital costs are shown in the following Table 21-8 and Table 21-9, including direct costs, indirect costs, and contingency.

Table 21-8: Initial Capital

Description	Total Cost
DIRECT COSTS – INITIAL	
AREA 00 – Site Infrastructure	
▪ General Site Work	$1,488,668
▪ Water supply	$465,807
▪ Buildings	$2,874,211
▪ Power Supply – (Power Generation Plant and site distribution)	$2,727,133
▪ Construction consumables (for this area only)	$26,811
▪ Freight and Insurance (for this area only)	$575,969
Total Process Infrastructure Cost:	**$8,158,600**
PROCESS PLANT - DIRECT COSTS	
▪ Area 10 - Crushing	$10,901,878
▪ Area 15 - Agglomeration	$3,348,925
▪ Area 20 – Ore Stacking	$728,400
▪ Area 45 – Heap Leach	$1,164,713
▪ Area 50 – Carbon Adsorption and Solution Management	$4,452,689
▪ Area 55 Acid Wash, Carbon Stripping, Regeneration, and Electrowinning	$2,495,946
▪ Area 61 Detoxification	0
▪ Area 65 Reagent Mix / Storage	$626,315
▪ Area 67 Utilities	$1,177,919

Table 21-8: Initial Capital

Description	Total Cost
Total Direct Cost:	**$33,055,385**
INDIRECT COSTS- INITIAL	
▪ Engineering	$2,194,616
▪ Procurement	$509,000
▪ Construction Management	$1,260,680
▪ Training	$75,000
▪ Initial Fill	$327,390
▪ Startup	$169,000
▪ Spare Parts	$654,780
▪ Mobile Equipment	$360,000
Total Indirect:	**$5,550,465**
Total Direct and Indirect:	**$38,605,851**
Contingency:	**$7,721,170**
TOTAL:	**$46,327,021**

Table 21-9: Sustaining Capital

Description	Total Cost
DIRECT COSTS – FUTURE	
Area 00 – Site Infrastructure	
General Site Work	n/a
Water supply	n/a
Buildings	n/a
Power Supply (Power Generation Plant and site distribution)	n/a
Construction consumables (for this area only)	n/a
Freight and Insurance (for this area only)	n/a
Total Process Infrastructure Cost	n/a
PROCESS PLANT – DIRECT COSTS	
Area 10 - Crushing	n/a
Area 15 - Agglomeration	n/a
Area 20 – Ore Stacking	n/a
Area 45 – Heap Leach	n/a
Area 50 – Carbon Adsorption and Solution Management	n/a
Area 55 Acid Wash, Carbon Stripping, Regeneration, and Electrowinning	$227,304
Area 61 Detoxification	$2,145,990
Area 65 Reagent Mix / Storage	$1,066,766
Area 67 Utilities	n/a
Total Direct Cost	$3,440,060
INDIRECT COSTS – FUTURE	
Engineering	n/a
Procurement	$27,500

Table 21-9: Sustaining Capital

Description	Total Cost
Construction Management	n/a
Training	n/a
Initial Fill	n/a
Startup	n/a
Spare Parts	n/a
Mobile Equipment	n/a
Total Indirect	$27,500
Total Direct And Indirect	$3,467,560
Contingency	$693,512
TOTAL	**$4,161,072**

21.1.10.1 Capital Cost Basis for Processing Plant

The direct costs exhibited in this estimate include infrastructure, buildings, materials and equipment, and the associated installation labour for the construction activities listed below:

- Crushing plant, process plant and power plant site development or rehabilitation.
- Agglomerated ore stockpile, truck haul and bull dozer on pad to level heap leach ore.
- Reagents' storage and mixing facilities.
- Utility equipment.
- Fresh water supply system, including power supply.

It is assumed that construction equipment and staff access roads will be available during scheduled construction periods on a continuous basis. Allowances are not included in the estimate for stand-by time or inefficiencies resulting from work or interferences initiated by others.

The process plant general arrangement depicts CMU/concrete walls at the refinery area; however, chain link fence is used within the estimate instead.

21.1.10.2 Direct Costs

The direct capital costs were based on the following list of items and documents:

- Field site visits;
- Preliminary design criteria and mass balance;
- Preliminary equipment list;
- Budget quotations for major equipment; and
- SGS equipment database for minor equipment.

Engineering drawings prepared for Brewery Creek by SGS and are listed in Table 21-10.

Table 21-10: Engineering Drawings

Drawing No.	Description
	Flowsheets
00-F-001	Overall Flowsheet Heap Leach Plant
00-F-002	Mass Balance
10-F-001	Flowsheet Crushing
15-F-001	Flowsheet Agglomeration
45-F-001	Flowsheet Solution Management
50-F-001	Flowsheet Carbon Adsorption
55-F-001	Flowsheet Stripping & Refining
55-F-002	Flowsheet Acid Washing & Carbon Reactivation
61-F-001	Flowsheet Heap Detoxification
65-F-001	Flowsheet Reagents
65-F-002	Flowsheet Reagents
67-F-001	Flowsheet Water
	Civil
05-G-010	Site Plan Overall View
05-G-011	**Site Plan ADR Plant Area**
05-G-012	Site Plan Crushing & Agglomeration
	General Arrangements
10-L-001	General Arrangement Crushing
10-L-015	General Arrangement Crushing Sections
10-L-016	General Arrangement Crushing Sections
10-L-017	**General Arrangement Crushing Sections**
10-L-018	General Arrangement Overland Conveyor Section
23-L-010	General Arrangement Heap Detoxification Plant Plan
23-L-015	General Arrangement ADR/Refinery Plant Plan
23-L-016	General Arrangement ADR Plant Refinery Upper Plan
23-L-020	General Arrangement ADR Plant Section
23-L-021	General Arrangement ADR Plant Section
95-L-001	General Arrangement Brewery Creek Main Substation
	Electrical Single Line Diagrams
00-E-001	Overall Power Supply & Distribution
10-E-001	Primary Crushing
15-E-001	Agglomeration and Lime Addition
45-E-001	Carbon Adsorption and Heap Leach
55-E-001	Acid Wash, Carbon Stripping, Regeneration and Electrowinning
61-E-001	**Detoxification and Utilities**
65-E-001	Reagent Mix/Storage

21.1.10.3 Equipment Costs

An equipment list was developed and incorporated into the cost estimate. The estimate for equipment costs were obtained from the following sources:

- Written or e-mailed budgetary estimates from vendors for major equipment.
- Historical data and budget costs of similar projects for miscellaneous equipment.

Major equipment quotations were obtained from North American equipment vendors. The cost estimate will reflect the result of equipment quote review where multiple quotes where received.

21.1.10.4 Construction Consumables for Process Plant

Construction consumables are included using a rate of 7 % of labour costs.

21.1.10.5 Freight and Equipment Insurance

Freight is included using a rate of 8 %of the equipment and bulk material costs.

21.1.10.6 Indirect Costs

Certain indirect costs exhibited in this estimate include, but are not limited to, labour, equipment and materials for the detailed activities set forth below:

- Feasibility Study - Includes an allowance for development of the project at a bankable feasibility level.
- Detailed Engineering - Includes estimated time, costs, and generated documents for the project to move into procurement and it allows a contractor to install the facilities.
- Procurement - Includes efforts for placement of RFQ's and bid documents, as well as coordination through delivery and storage on-site.
- Construction Management - Includes management and scheduling of the construction effort. Future construction for the detoxification and overland conveyor system will be carried by the Owner's on-site engineering staff and is to be included within the Owner's cost. This item is not included in the capital cost estimate.
- Training – includes an allowance for training personnel to operate the process plant.
- Initial Fill – includes estimate at 2.5% of purchased equipment cost (uninstalled).
- Start-up – includes an allowance for process engineers and vendor equipment technicians for commissioning and start-up of the process plant.
- Spare parts – includes an allowance of 5% of the equipment costs.

21.1.10.7 Mobile Equipment for Construction

This section includes the estimate for equipment needed for operations and to transport tools, materials, and other items for maintenance of the crushing and process plant. Table 21-11 shows the mobile equipment included in the estimate.

Table 21-11: Mobile Equipment

Item	Total No.
988 Wheel Loader	1
D8 Dozer	1
Mobile Crane - 50 Ton, all terrain	1
Forklift	2
All Terrain Forklift	1

21.1.11 Exclusions for Processing

SGS has excluded the following items from the capital cost estimate. Owner's costs include:

- Exploration;
- Site Water Treatment Systems;
- Storm Water Diversion;
- Permits, royalties and licenses;
- Environmental testing and monitoring;
- Metallurgical testing;
- Escalation;
- Insurance;
- Taxes, duty and import fees;
- Geotechnical design and facility costs; and
- Allowance for design growth or specification changes.

21.1.12 Heap Leach Facility Construction Capital Costs

The estimated cost of construction of cells 8 to 10 is summarised in Table 21-12. These costs are an average of roughly $85 per square metre for the new heap leach cells.

Table 21-12: Heap Leach Cells 8 to 10 Construction Costs

Heap Leach Pad		Total Cost
1.1 Grading and Liner System		**$7,458,609**
1.1.1	Clearing and grubbing	$144,400
1.1.2	Topsoil Stripping	$291,753
1.1.3	Pad Grading (cut) (incl active-permafrost)	$2,037,495
1.1.4	Pad Grading (fill)	$1,461,323
1.1.5	Subgrade preparation (fill)	$566,834
1.1.6	Compacted Silt	$586,112

Table 21-12: Heap Leach Cells 8 to 10 Construction Costs

Heap Leach Pad		Total Cost
1.1.7	Primary Geomembrane - top liner (1.0 mm PVC)	$1,642,552
1.1.8	Ore Cushion Material/overliner Drain Fill	$728,140
1.2 Leak Detection and Removal System		**$497,814**
1.2.1	Collector Pipe (50mm ADS N-12 perforated)	$236,455
1.2.2	Header Pipe (100mm ADS N-12 perforated)	$84,455
1.2.3	Primary Header Pipe (100mm solid HDPE)	$46,204
1.2.4	Header Pipe Through Embankment (300mm carbon steel)	$10,698
1.2.5	100mm x 50mm HDPE reducers	$90
1.2.6	100mm N-12 Perforated 10 100mm solid HDPE coupler	$30
1.2.7	100mm x 100mm Wye	$300
1.2.8	French Drain (granular fill drainage material	$22,680
1.2.9	Geotextile	$96,903
1.3 Solution Collection System		**$3,107,321**
1.3.1	Collector Pipe (100mm ADS N-12 perforated)	$2,658,347
1.3.2	Collection Header Pipe (375mm ADS N-12 perforated)	$356,319
1.3.3	Header pipe through embankment (400mm carbon steel)	$41,828
1.3.4	Header pipe through embankment (600mm carbon steel)	$41,828
1.3.5	Liner penetration transition structure	$9,000
1.4 New Solution Collection Ditch (Adjacent to Embankment)		**$25,987**
1.4.1	Solution ditch compacted silt	$4,252
1.4.2	solution ditch geomembrane	$21,168
1.4.3	solution ditch granular pipe bedding	$567
2.1 Pregnant Ponds		**$405,066**
2.1.1	Pregnant pond grading (fill)	$60,595
2.1.2	subgrade preparation (fill)	$83,783
2.1.3	compacted silt (soil liner)	$17,832
2.1.4	secondary (bottom) geomembrane (1.0mm HDPE)	$82,177
2.1.5	geonet drain liner for leak detection system	$74,547
2.1.6	primary (top) geomembrane (2mm HDPE)	$86,132
2.2 Barren Pond		**$422,333**
2.2.1	Pond grading (cut)	$82,319
2.2.2	subgrade preparation (fill)	$83,783
2.2.3	compacted silt (soil liner)	$13,374
2.2.4	secondary (bottom) geomembrane (1.0mm HDPE)	$82,177
2.2.5	geonet drain liner for leak detection system	$74,547
2.2.6	primary (top) geomembrane (2mm HDPE)	$86,132
2.3 Overflow Pond		**$770,868**
2.3.1	Pond grading (cut)	$353,833
2.3.2	subgrade preparation (fill)	$168,810

Table 21-12: Heap Leach Cells 8 to 10 Construction Costs

Heap Leach Pad		Total Cost
2.3.3	compacted silt (soil liner)	$74,677
2.3.4	geomembrane (2.0mm HDPE)	$173,548
3.1 Diversion Channel		**$41,782**
3.1.1	Clearing and Grubbing	$20,900
3.1.2	Topsoil Stripping	$16,830
3.1.3	Cut	$4,052
3.2 Monitoring Vault		**$34,477**
3.2.1	Pump	$3,125
3.2.2	Sampling port	$ -
3.2.3	Knifegate valve	$3,700
3.2.4	Flow meter	$12,000
3.2.5	Concrete box	$11,200
3.2.6	300mm riser	$576
3.2.7	300mm tees	$1,376
3.2.8	300mm pipe	$2,500
Total cost of construction for cells 8 to 10		**$12,764,256**

21.2 Operating Costs

Brewery Creek site operating costs have been estimated for mining, processing and general and administrative (G & A) costs. The following inputs costs or parameters have been used to estimate operating costs:

- Diesel cost of $1.19/litre delivered to site.
- Propane cost of $0.83/litre delivered to site.
- Effective LNG produced energy cost of 20c/kWhr.
- Yukon Energy Corp. supplied power at $ 0.114/kWhr.
- Operating days for mining, crushing and heap leaching stacking of 230 days per annum.
- Solution management for 365 days per annum.
- Contractor markup of 30% where applicable.

A summary of all operating costs are shown in Table 21-13.

Table 21-13: Summary of Base Case Operating Cost Estimates for the Brewery Creek Operation

Costs			
Item	**Cost in USD$**	**Units**	**Source**
LOM average cost of mining per tonne process feed[1]	$13.36	$/tonne	Modelled using Runge Xeras™
LOM average cost of mining process feed from old heap leach	$1.17	$/tonne	Modelled using Runge Xeras™
LOM average cost of mining process feed in the pits[1]	$3.52		
LOM average cost of mining waste rock in pits[1]	$2.61	$/tonne	Modelled using Runge Xeras™
LOM average processing costs and placement on heap leach pad	$8.41	$/tonne	SGS and modelled using Runge Xeras™
General and administrative costs per tonne process feed	**$3.11**	$/tonne	Estimated for each year of operation

[1] Includes cost of mining equipment lease which averages US$ 0.59/ tonne rock mined in the pits

21.2.1 General and Administrative Costs

The LOM G and A costs have been estimated to be the equivalent of CAD$3.39 per tonne of rock processed. The average cost of G and A during pit mining operations is CAD$5.1 million per year. Table 21-14 shows the breakdown of the base case G and A.

G and A includes salaries for management and administrative personnel, mining planning and general site maintenance. The general and administrative costs also include office and camp operating costs, but exclude electricity costs, which fall under processing.

Table 21-14: Breakdown of G and A Cost Estimate for Brewery Creek over LOM

G and A costs in CAD$000		Total LOM	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Total G & A labour	Salaried staff [1]	$21,009	$647	$1,850	$2,263	$2,263	$2,263	$2,263	$2,263	$2,263	$2,263	$1,958	$714
Camp costs	Camp operation @ $60 pp	$8,701	$91	$1,417	$902	$943	$971	$929	$860	$888	$916	$784	$
	Camp maintenance	$727	$73	$73	$73	$73	$73	$73	$73	$73	$73	$73	
Transport	Management Dawson to site	$955	$71	$92	$92	$92	$92	$92	$92	$92	$92	$92	$55
Services	Legal	$1,042	$	$110	$110	$110	$110	$110	$110	$110	$110	$110	$55
	Consultants	$798	$	$84	$84	$84	$84	$84	$84	$84	$84	$84	$42
	Reclamation services	$642	$	$	$80	$80	$80	$80	$80	$80	$80	$80	
	Environmental non-labour	$83	$	$	$10	$10	$10	$10	$10	$10	$10	$10	

Table 21-14: Breakdown of G and A Cost Estimate for Brewery Creek over LOM

G and A costs in CAD$000		Total LOM	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
	Professional development and training	$1,467	$15	$276	$147	$156	$162	$153	$138	$144	$150	$126	
	Accounting	$530	$48	$48	$48	$48	$48	$48	$48	$48	$48	$48	$48
	Waste disposal / collection	$591	$30	$60	$60	$60	$60	$60	$60	$60	$60	$60	$20
	Contract environ. services	$1,785	$	$178	$178	$178	$178	$178	$178	$178	$178	$178	$178
Taxes and insurance	Property taxes	$1,449	$132	$132	$132	$132	$132	$132	$132	$132	$132	$132	$132
	Insurance	$3,683	$125	$374	$374	$374	$374	$374	$374	$374	$374	$374	$187
Supplies	Engineering supplies	$287	$14	$29	$29	$29	$29	$29	$29	$29	$29	$29	$14
	Office supplies	$859	$43	$86	$86	$86	$86	$86	$86	$86	$86	$86	$43
	Safety supplies and training	$564	$28	$56	$56	$56	$56	$56	$56	$56	$56	$56	$28
	Propane	$194	$18	$18	$18	$18	$18	$18	$18	$18	$18	$18	$9
Other	Donations	$406	$	$45	$45	$45	$45	$45	$45	$45	$45	$45	
	Communication systems	$1,063	$106	$106	$106	$106	$106	$106	$106	$106	$106	$106	
	Employee physicals	$335	$3	$59	$34	$36	$37	$35	$32	$34	$35	$29	
	Employee relations	$240			$30	$30	$30	$30	$30	$30	$30	$30	
	Employee transport [3]	$1,533		$	$182	$192	$198	$189	$173	$179	$186	$157	$77
Total G & A		$48,943	$1,444	$5,094	$5,141	$5,203	$5,244	$5,182	$5,079	$5,121	$5,162	$4,668	$1,603

[1] Salaried staff includes a salary burden of 30%
[2] The assumption is that the owner will provide transport for some managerial and administrative day staff from Dawson City every day
[3] Transport costs are based of 50% of permanent staff traveling from further than Dawson City at $800 per round trip

The salaries used in estimating the G and A costs are shown in Table 21-15.

Table 21-15: Salaries Used in G and A Calculation

G and A Salaries (including burden of 30%)	
Staff member	**Cost to company in CAD$**
General Manager	$240,500
Technical services manager/Assistant GM	$175,500
Human Resources	$136,500
Administrative Assistant	$52,000
Mine Manager	$175,500
Service Vehicle mechanic	$84,500
Environmental Manager	$133,900

Table 21-15: Salaries Used in G and A Calculation

G and A Salaries (including burden of 30%)	
Staff member	**Cost to company in CAD$**
Environmental Assistant	$71,500
Geology & Ore control	$127,400
Engineering & Planning	$127,400
Site electrician	$84,500
Surveying	$97,500
Warehouse/Purchasing	$68,900
Site Accountant	$68,900
Health, Safety, Security Manager	$71,500
Training manager	$71,500
Security guards	$39,000
Health & Safety assistants	$71,500

21.2.2 Mining Costs

Mining costs have been modelled based on the cost of operating mining equipment, diesel and explosive consumption, and estimated equipment productivities as shown in Table 16-17. The mining schedule, as discussed in section 16.5.3 was then used to estimate the hours of machine use per year required. These hours as well as equipment specifications were used as inputs into Runge XERAS™, which uses factoring of the equipment capital cost and fuel consumption to estimate maintenance labour and parts costs, lubricant costs, overhauling costs and costs for wear parts.

A summary of the mining grouped by allocation and by activity is shown in Table 21-16.

Table 21-16: Breakdown of Mining Costs By Allocation and By Activity

Operating costs in USD$000	Year - 2	Year - 1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Labour Cost										
Operators 5 Day	$	$763	$1,058	$1,058	$1,058	$1,061	$1,058	$1,058	$1,058	$1,051
Operators 7 Day	$33	$2,145	$2,401	$2,427	$2,364	$2,416	$1,920	$2,251	$2,335	$1,640
Maintenance	$43	$1,839	$2,029	$2,135	$1,990	$2,052	$1,754	$2,071	$2,064	$1,415
Technical Staff	$	$41	$83	$83	$83	$83	$83	$83	$83	$83
Professional Staff	$83	$166	$165	$165	$165	$166	$165	$165	$165	$166
Total Labour Costs	**$159**	**$4,954**	**$5,735**	**$5,867**	**$5,659**	**$5,777**	**$4,979**	**$5,628**	**$5,705**	**$4,354**
Equipment costs										
Liquid Fuels	$46	$3,009	$3,422	$3,375	$3,322	$3,390	$2,733	$3,299	$3,443	$2,221
Lube	$7	$390	$469	$451	$463	$460	$374	$437	$477	$310
Tyre Replacement	$	$273	$261	$275	$235	$268	$215	$300	$282	$174
Ground Engaging Tools	$11	$300	$312	$350	$347	$322	$297	$290	$330	$210
Mechanical Repair Parts	$15	$643	$807	$782	$799	$778	$597	$737	$803	$525
Overhaul	$2	$342	$459	$477	$468	$470	$334	$433	$451	$291
Total Equipment Costs	**$80**	**$4,956**	**$5,730**	**$5,711**	**$5,636**	**$5,689**	**$4,550**	**$5,496**	**$5,786**	**$3,731**

Table 21-16: Breakdown of Mining Costs By Allocation and By Activity

Operating costs in USD$000	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Other										
Explosives	$	$1,740	$1,496	$1,765	$1,725	$1,652	$1,660	$1,474	$1,825	$1,146
Other Operating Costs	$	$423	$422	$422	$422	$423	$422	$422	$422	$423
Total other	**$**	**$2,163**	**$1,918**	**$2,187**	**$2,147**	**$2,075**	**$2,082**	**$1,896**	**$2,247**	**$1,569**
Total Site Operating Costs	**$239**	**$12,073**	**$13,383**	**$13,766**	**$13,442**	**$13,541**	**$11,611**	**$13,020**	**$13,738**	**$9,655**
Costs Breakdown by activity										
Cost of mining process feed			$3,583	$1,819	$3,770	$2,937	$1,944	$1,469	$3,563	$2,783
Cost of mining waste rock	$157	$8,221	$5,048	$7,109	$4,912	$6,123	$6,084	$7,825	$6,687	$4,045
Service equipment and mining admin.	$83	$2,910	$3,137	$3,129	$3,141	$3,148	$3,153	$3,141	$3,139	$2,617
Haul road maintenance costs	$	$	$451	$455	$456	$303	$299	$299	$299	$184
Cost of mining material from leach pad	$	$942	$1,156	$1,219	$1,156	$997	$124	$221	$44	$24

21.2.3 Mining Equipment Leasing

For the PEA mining equipment leasing has been considered to reduce initial capital. The total purchase cost including contingency and freight has been estimated at US$17.8 million. Leasing rates have been estimated as a repayments including 18% interest, at US$3.5 million per year.

21.2.4 Heap Leach Stacking Costs (Mobile Machinery Option)

In addition to mining costs and processing costs, a cost has been estimated for the placement of heap leach feed onto the heap leach pad. It is estimated that it will cost $0.79 per tonne placed on the heap leach pad.

This has been estimated based on the use of mobile machinery in the form of front end loaders, haul trucks and a dozer to spread the material on the heap leach pad. Runge XERAS™ has been used to generate this estimate, based on estimated equipment productivities and haul distances.

21.2.5 Processing Plant Operating Costs

This operating cost estimate (OPEX) is provided for the Brewery Creek mine to satisfy the requirements of the NI 43-101 report. Operating costs for the process plant are a major portion of the overall cost of operating the heap leach facility. This section describes the operating costs associated with the process facility as described in Section 17 of this report.

Operating costs for the Brewery Creek heap leach facility are based on the process design criteria and the process flow design. Operating costs include:

- Consumables;
- Power and Energy;
- Labor; and

- Wear parts.

OPEX for Brewery Creek is shown using Owner Operated Crushing. Operating cost is based on budgetary quotations solicited from likely vendors, or database costs for similar operations in the Yukon. Active crushing and placement occurs 230 days each year and shuts down during the winter months. Some activities continue to occur, including solution management.

Consumables for Brewery Creek include reagents, and fuel for unit operations and heating. Costs are shown in Table 21-17.

Table 21-17: Estimated Reagent and Fuel Costs

Reagents	
Reagent Prices - US$/kg - FOB to Site	**US$/kg**
Anti-scalant	$2.25
Sodium Hydroxide	$1.17
Activated Carbon	$2.57
Hydrochloric Acid	$1.80
Sodium Cyanide Briquettes	$3.33
Lime	$0.44
Cement	$0.41
Fuels – $/l	**US$/l**
Diesel	$1.11
Propane	$0.73

Costs are based on budgetary quotations for delivery to the Brewery Creek site or estimated based on previous projects based in the Yukon. Lime costs include reagent quote FOB from Langley, BC and shipping costs from Langley to the plant site. Lime is delivered in 1-tonne bags as bulk trucking is not available. Cement costs are based on 40-tonne loads. Anti-scalant costs are from data base and assume shipping from Vancouver, BC. Fuel costs for both diesel and propane were provided by Brewery Creek.

Full scale operations are anticipated in the first year. Reagent consumptions for lime, cement, and cyanide are based on test work as shown in section 13.0, and from historical data. Other reagents, anti-scalant, hydroxide, and carbon are based on historical operating data from the original monthly reports from Viceroy Minerals Corporation. Reagent consumptions are based on a nominal tonnage of 1,725,000/year. Table 21-18 show the estimated reagent costs for the each year based on mining schedule due to different reagent consumption rate for different ore type.

Table 21-18: Reagents

Reagent	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Anti-scalant	180,792	180,478	189,057	180,687	180,583	180,374	180,583	135,489	**1,408,043**
Sodium Hydroxide	90,979	90,821	95,139	90,927	90,874	90,769	90,874	68,182	**708,564**
Activated Carbon	88,957	88,803	93,024	88,906	88,854	88,752	88,854	66,667	**692,818**
Hydrochloric Acid	62,208	62,100	65,052	62,172	62,136	62,064	62,136	46,620	**484,488**
Sodium Cyanide Briquettes	1,439,659	1,527,538	1,847,401	2,015,033	1,737,131	2,259,728	2,036,348	1,611,087	**14,473,925**
Lime	1,616,936	861,153	2,157,208	1,867,522	1,843,006	1,002,230	1,951,944	1,528,780	**12,828,778**
Cement	708,953	2,429,393	409,961	858,296	1,364,101	2,377,775	712,962	434,565	**9,296,006**
TOTAL (US$)	**4,188,484**	**5,240,286**	**4,856,843**	**5,163,543**	**5,366,685**	**6,061,691**	**5,123,701**	**3,891,390**	**39,892,621**
$/t	**2.42**	**3.04**	**2.69**	**2.99**	**3.11**	**3.52**	**2.97**	**3.00**	**2.96**

Diesel fuel is required for operating the carbon regeneration kiln and refinery furnace, and propane is used to heat the ADR facility and ancillary facilities year-round. Table 21-19 show the breakdown of estimated fuel for heating.

Table 21-19: Fuel

Fuel	Type	$/l	l/day	US$/Year	US$/t
Propane	Bullet	0.73	1,196	318,154	0.18
Additional Diesel - ADR	Bulk	1.11	477	60,713	0.04
TOTAL				**378,867**	**0.22**

The largest power user is the crushing plant. Crushing and placement of materials occur only during the operating season, 230 days each year, but solution management continues throughout the year. Power consumption is classified according to days when the site is fully operational and during the winter months when power consumption is reduced. Table 21-20 shows power costs per tonne of ore processed for owner the operated crushing option.

Table 21-20: Power

	Days	kW	kWh/day	US$/Year	US$/t
Process Power Draw	230	3,364	80,736	4,122,247	2.39
Heap Leach - Winter Ops	135	375	8,993	269,523	0.16
ADR - Winter Ops	135	1,159	27,822	833,790	0.48
TOTAL				**5,225,560**	**3.03**

This equates to roughly 13.65 kWhrs per tonne of process capacity.

Labour costs include salary, benefits, and overtime for hourly employees. Labour costs assume that personnel required to winter-over will be salary personnel and ADR plant operators. Annual dollars reflect the scheduled days for full operation and additional dollars needed for employees required year-round. Table 21-21 show the labour cost breakdown for process plant.

Table 21-21: Labour

Description	Crew Size	US$/Year	US$/t
Salary	4	468,000	0.27
Crusher	12	703,659	0.41
Heap Leach	8	411,457	0.24
ADR	10	875,703	0.51
Laboratory	4	200,265	0.12
Maintenance	6	412,402	0.24
TOTAL	**44**	**3,071,487**	**1.78**

Consumables for the heap leach operations include dripper piping, maintenance parts for the ADR plant and crushing plant, and the assay lab costs. Dripper length is estimated based on a 24 inch centre. Costs are FOB to site and include the 4-inch lay-flat hose, needed to connect the drip emitters to the main solution line.

Assay costs are estimated by assuming a number of samples processed by the laboratory, each requiring multiple determinations, and assigning a cost for each determination. Samples include blast-hole, stockpile, and process solutions and carbon.

Maintenance and wear parts for the owner-operated crusher use a standard assumption for costs, based on the capital cost of the original equipment.

Table 21-22 summarizes the estimated costs for other consumables.

Table 21-22: Other Consumables

	Days	Det./tonne	US$/Det.	US$/Year	US$/t
Assays	230	0.027	4.00	184,000	0.11
Wear Parts	**% of CAPEX**	**CAPEX**		**US$/Year**	**US$/t**
Crusher	5.0	5,942,653		297,133	0.17
ADR	1.0	5,227,609		52,276	0.03
Heap Leach	**$/m**	**total meters**		**US$/YEAR**	**US$/t**
Dripper	5.30	22,100		117,130	0.07
TOTAL				**650,539**	**0.38**

Table 21-23 shows the summarized annual operating costs for the processing plant.

Table 21-23: Processing Plant Annual Operating Costs

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Reagents	4,188,484	5,240,286	4,856,843	5,163,543	5,366,685	6,061,691	5,123,701	3,891,390	39,892,621
Fuel	379,526	378,867	396,877	379,306	379,086	378,647	379,086	284,425	2,955,820
Power	5,234,648	5,225,560	5,473,964	5,231,619	5,228,589	5,222,531	5,228,589	3,922,957	40,768,457
Labor	3,076,829	3,071,487	3,217,494	3,075,048	3,073,267	3,069,706	3,073,267	2,305,841	23,962,939
Other	651,670	650,539	681,463	651,293	650,916	650,162	650,916	488,375	5,075,334

Table 21-23: Processing Plant Annual Operating Costs

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
TOTAL (US$)	13,531,156	14,566,738	14,626,640	14,500,809	14,698,544	15,382,737	14,455,561	10,892,987	112,655,171
$/t	**7.83**	**8.44**	**8.09**	**8.40**	**8.52**	**8.92**	**8.38**	**8.41**	**8.37**

22.0 ECONOMIC ANALYSIS

22.1 Introduction

Tetra Tech EBA prepared an economic evaluation of the Brewery Creek Project using discount cash flow modelling. The project with 9 years of operating life as proposed in the PEA has positive economics. Key economic modelling results are shown in Table 22-1.

Table 22-1: Summary of Economic Modelling Results

Summary of financial results in US$'000	
Using a Gold price of US$ 1,250 / oz. troy	
Pretax and royalty $NPV_{5\%}$	$45,658
Pretax and Royalty IRR	22%
Post Tax and Royalty $NPV_{5\%}$	$23,315
Post Tax and Royalty IRR	15%
Payback period	3.2 years
Using a Gold price of US$ 1300 / oz.	
Pretax and royalty $NPV_{5\%}$	$59,431
Pretax and Royalty IRR	27%
Post Tax and Royalty $NPV_{5\%}$	$32,315
Post Tax and Royalty IRR	19%
Payback period	2.9 years

The economic analysis is based on the extraction of indicated and inferred resources from both open pits and heap leach resources. No financing arrangements have been included in the financial model, except for consideration of leasing mining equipment, which is included as an operating expense. The remaining capital is assumed to be 100% equity based.

Tetra Tech EBA's parameters used to assess the feasibility of the project, as of August 2014, and as used in the base case were as follows:

- Price of gold for base case – US$1,250/oz
- Currency exchange – CAD $1.09 = US$1
- Reclamation cost of US$ 8 million in line with reclamation fund provision
- Staged Yukon Royalty
- Private Royalties as detailed in Section 4.3

- Yukon and Federal tax each at 15% less allowable deductions
- Gold freight and marketing costs of totalling US$4 per ounce Dore
- Smelter deduction of 0.5%
- Discount rate of 5%
- Canadian Federal tax rate of 15%. Yukon Territory tax rate of 15%

Capital and operating costs have been modelled for the life of mine including two preproduction years. Working capital has been provided to cover operating costs for the first quarter of operations during the first year and monthly thereafter. Working capital is adjusted annually based on the dollar amount required. Since working capital is provided for operating expenses, which are deducted from revenues annually in the model, the funds provided for working capital are recovered over the life of mine. Table 22-2 summarises the undiscounted production and economic results of the financial model.

Table 22-2: Summary of Undiscounted Production and Economic Results

Summary of operations		
Total tonnes processed	14,445,179	Tonnes
Average grade of processed tonnes	1.18	g/t
Total Ounces Mined	548,648	Oz.
Total Ounces Recovered	372,333	Oz.
Total Dore Ounces produced	413,703	Oz.
Total ounces paid for	370,471	Oz.
Total revenue received	$463,089	US$000
Selling costs	$1,531	US$000
Total operating costs	$288,232	US$000
Operating margin	$173,326	US$000
Total capital costs including sustaining and contingency	$89,410	US$000
Reclamation costs	$8,000	US$000
Net cash flow before taxes and royalties	$77,043	US$000
Total Royalties	$16,342	US$000
Total Taxes	$14,864	US$000
Total cash flow after taxes and royalties	$46,578	US$000

22.2 Taxes and Royalties

Taxes have been included for both the Yukon Territories and Canadian Federal tax. Yukon Royalties and private royalties have been deducted from post-tax financial results. Yukon Royalties are applied as a staged royalty as per Yukon government guidelines. Development costs are deductible from Yukon Royalty estimates. Private royalties include Till Capital at 0.5 % and Alexco at 2% of operating cash flow. The remaining ounces pertaining to the Franco-Nevada Royalty of 21,516 oz. have been included at $40/oz. This royalty will be taken from ounces mined at Fosters in year 1. Energold Royalties have been estimated based on 781 claims relevant to the Energold Royalty, using 5% of net profits less allowable deductions.

Yukon Taxes are included at 15% less allowable deductions which include Canadian Exploration Expenses (CEE) and deprivation allowances. Federal Taxes are applied to the taxable income after deduction of Yukon Taxes and are applied at 15%.

Table 22-3 is a summary of the Royalties and Taxes included in the financial model.

Table 22-3: Summary of Taxes and Royalties included in the Financial Model

Taxes and Royalties	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Yukon Royalties	$3,635	$	$	$805	$542	$520	$	$568	$1,139	$61
Till Capital at 0.5% NSR	$2,315	$473	$225	$341	$266	$248	$183	$274	$237	$68
Alexco at 2% of NSR	$9,189	$1,890	$900	$1,349	$1,054	$982	$732	$1,083	$926	$272
Franco-Nevada	$861	$861	$	$	$	$	$	$	$	$
Energold	$342	$277	$	$64	$	$	$	$	$	$
Yukon Taxes at 15% less deductions	$8,035	$	$	$1,705	$1,248	$1,212	$	$1,294	$2,391	$184
Federal Taxes at 15% less deductions	$6,829	$	$	$1,450	$1,061	$1,030	$	$1,100	$2,032	$156
Total Taxes and Royalties	$31,206	$3,501	$1,125	$5,714	$4,171	$3,993	$914	$4,320	$6,725	$742

22.3 Base Case Sensitivity Analysis

Tetra Tech has completed sensitivity analyses of the project post tax $NPV_{5\%}$, for capital costs, operating costs and gold price. The project is found to be most sensitive to gold price. Figure 22–1 shows the results of the sensitivity analysis. The sensitivities were generated using a multiplier on the attribute being evaluated from 0.7 to 1.3, in in increments of 0.1.

Figure 22–1: Sensitivity Chart for Post-Tax NPV

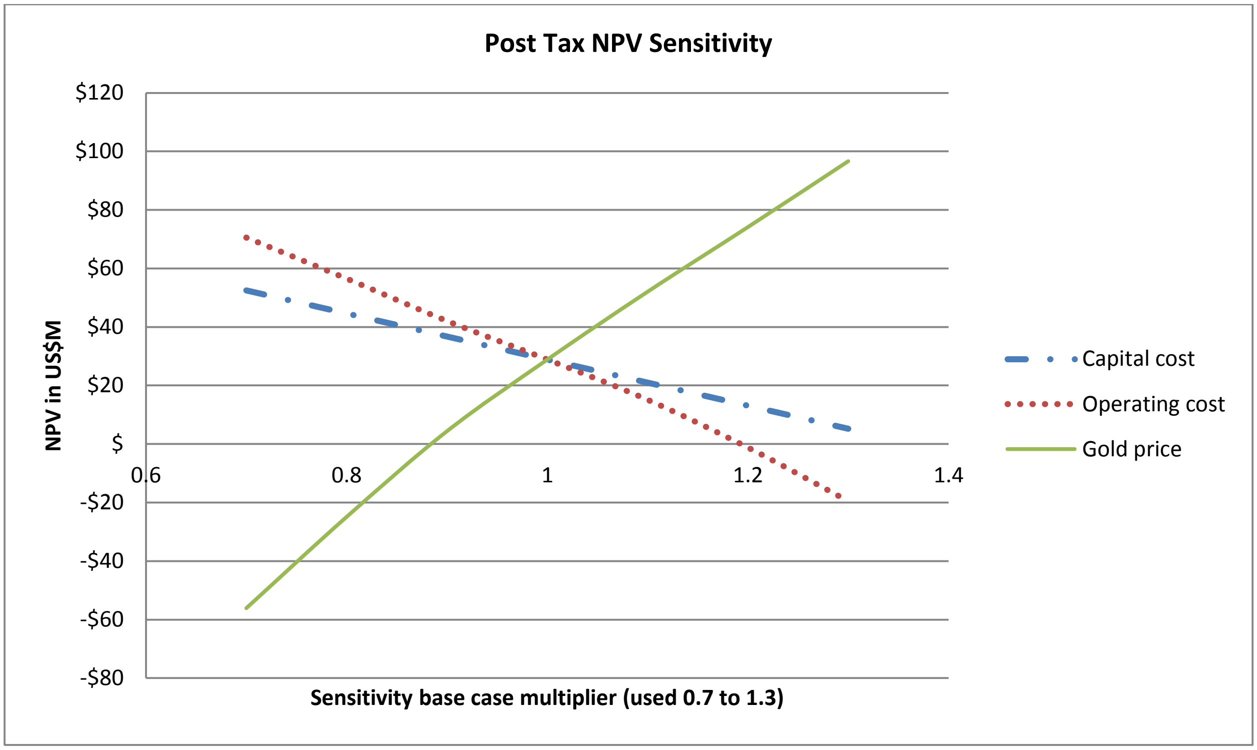


Table 22-4 shows the base case sensitivities at various gold prices. The break even gold price is roughly $1,135 per troy ounce.

Table 22-4: Post Tax and Royalty Sensitivities for Various Gold Prices

Gold Price in US$	NPV in US$000	IRR
$1,100	-$7,610	1%
$1,150	$4,001	7%
$1,250	$23,315	15%
$1,375	$46,858	24%
$1,500	$69,360	32%

22.4 Results of Trade-Off Studies

Trade-off studies were evaluated against the base case selected for the project were undertaken. These include owner purchase of all mining equipment and the use of contract miners, as opposed to buying all mining equipment. These were evaluated against the base case which considers leasing of all mining equipment. The results of the trade-off studies are presented in Table 22-5. Owner mining has been estimated to have the best post-tax results for the project, however at a higher capital cost. This result is also subject to leasing rates obtained for leased equipment. Contract mining has not been found to be favourable due to increased mining costs.

Table 22-5: Results of the Trade-Off Studies

Results of trade-off studies	Post tax NPV in US$000	Post Tax IRR
Base case (Equipment leasing)	$23,315	15%
Owner mining	$22,690	13%
Contract mining	$10,812	9%

22.5 Discount Cash Flow

The pre and post-tax discounted cash flow used as basis for the financial modelling is included in Table 22-6.

Table 22-6: Results of the Technical Economic Analysis

Brewery Creek Preliminary Economic Assessment

Description	Units	Total	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Base case													
Estimate of Income					>> Begin production								End <<
Production Summary (Production from Open Pit)													
Process feed from pits in schedule	t	10,264,483			1,625,815	839,794	1,736,022	1,459,782	1,164,814	725,513	1,525,873	1,186,870	
Processed from old leach pad in season	t	3,195,286			102,425	885,738	71,134	267,379	561,496	999,487	199,667	107,960	
RoM Ore to Process	t	14,445,179		0	1,728,240	1,725,532	1,807,156	1,727,161	1,726,310	1,725,000	1,725,540	1,294,830	985,410
Waste rock mined from pits	t	43,519,880		6,539,449	3,770,983	5,762,306	4,475,621	4,566,620	4,592,776	5,876,301	5,051,372	2,884,472	
Striping ratio		4.24			2.3	6.9	2.6	3.1	3.9	8.1	3.3	2.4	
Total tonnes mined	t	61,330,746		7,843,717	6,995,667	8,288,156	7,799,323	7,395,892	5,757,590	6,601,814	6,577,245	4,071,342	
Production Summary Gold Produced													
Contained Metal AS MINED													
Gold from pits as feed	g	13,845,462			3,159,921	1,100,911	2,234,543	1,770,279	1,319,926	767,215	1,856,273	1,636,394	
Gold from reprocessing old HL	g	3,219,136			78,867	682,018	54,773	205,882	432,352	769,605	153,744	83,129	758,766
Total contained metal	g	17,064,598			3,238,788	1,782,929	2,289,316	1,976,161	1,752,278	1,536,820	2,010,017	1,719,523	758,766
Contained Grade:													
Gold grade from pits	g/t	1.35			1.94	1.31	1.29	1.21	1.13	1.06	1.22	1.38	0.00
Gold from reprocessing old HL (Ind and Inf)	g/t	0.77			0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77
Overall process feed head grade	g/t	1.18			1.87	1.03	1.27	1.14	1.02	0.89	1.16	1.33	0.77
Recovered Metal from heap leach													
Gold grams from pits recovered	g	10,132,063			2,328,313	816,702	1,682,387	1,239,196	1,046,848	568,354	1,299,391	1,148,862	
Gold oz. from pits recovered	oz.	325,758		0	74,858	26,322	54,091	39,842	33,657	18,273	41,777	36,937	0
Gold oz. from reprocessing old heap leach recovered	oz.	46,575			1,141	9,867	792	2,979	6,255	11,135	2,224	1,203	10,978
Total ounces recovered	oz.	372,333			75,999	36,190	54,883	42,820	39,913	29,408	44,001	38,140	10,978
Ounces DORE produced	oz.	413,703			84,444	40,211	60,981	47,578	44,348	32,676	48,890	42,378	12,198
Metal Deductions													
Gold	0	1,807	0	0	390	181	274	214	200	147	220	191	55
Payable Metal													
Gold Spot Ounces Sold	Oz	370,471	0	0	75,619	36,009	54,609	42,606	39,713	29,261	43,781	37,949	10,923
Estimate of Cash Flow													
Market Prices			0										
Gold Spot in USD	1	$1,250			$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250
Net Revenue													
Gold Sales in USD	$000s	$463,089	$	$	$94,524	$45,011	$68,261	$53,258	$49,641	$36,576	$54,727	$47,437	$13,654
NSR USD	$000s	$463,089	$	$	$94,524	$45,011	$68,261	$53,258	$49,641	$36,576	$54,727	$47,437	$13,654
Freight	$1	$414			$84	$40	$61	$48	$44	$33	$49	$42	$12
Refining	$3	$1,117		$	$228	$109	$165	$128	$120	$88	$132	$114	$33
Gross Income USD	$000s	$461,972			$94,212	$44,862	$68,036	$53,082	$49,477	$36,456	$54,546	$47,280	$13,609
Mining costs in CAD$000s (unless noted)													
Direct mining cost leach feed reprocessing	$0.79	$5,964		$1,027	$1,260	$1,329	$1,260	$1,087					
Cost of sampling material for reprocessing	$0.25	$1,887		$326	$400	$422	$397	$342	$	$	$	$	
Rehandling material reprocessing material for processing	$0.24	$675	0	$	$	$	$	$	$135	$241	$48	$26	$225
Cost of mining heap leach in USD$	*$1.17*												
Direct mining cost leach feed from pits	$2.32	$23,836			$3,905	$1,983	$4,109	$3,202	$2,119	$1,601	$3,883	$3,034	
Direct mining cost waste	$1.41	$61,270	$171	$8,961	$5,503	$7,749	$5,354	$6,674	$6,632	$8,529	$7,288	$4,409	
Mine management, support and admin	$0.66	$29,992	$90	$3,172	$3,420	$3,411	$3,424	$3,431	$3,437	$3,424	$3,422	$2,852	
Haul road constr. And maint.	$0.06	$2,993			$492	$496	$497	$330	$325	$325	$325	$201	
Lease of mining equipment at 18% interest	$0.59	$31,603	$1,756	$3,511	$3,511	$3,511	$3,511	$3,511	$3,511	$3,511	$3,511	$1,756	
Mining cost per tonne leach feed from pits USD$	*$13.38*												
Processing cost leach feed	$8.37	$120,903			$13,531	$14,567	$14,827	$14,501	$14,699	$15,383	$14,456	$10,893	$8,248
Material haul to leach pad	$0.79	$11,479			$1,373	$1,371	$1,436	$1,372	$1,372	$1,371	$1,371	$1,029	$783
Average LOM process and place cost in USD$	*$8.41*												
		$											
G and A	$3.39	$49,033	$1,444	$5,094	$5,141	$5,203	$5,244	$5,182	$5,079	$5,121	$5,162	$4,668	$1,694
G and A USD$	*$3.11*												
Operating Costs in USD	$000s	$288,232	$	$	$35,354	$36,736	$36,568	$36,361	$34,229	$36,244	$36,209	$26,484	$10,046
Operating Costs per tonne leach feed USD		$19.95			$20	$21	$20	$21	$20	$21	$21	$20	$10
Operating Costs per oz. USD		$778			$468	$1,020	$670	$853	$862	$1,239	$827	$696	$920
Operating Cash-Flow USD	$000s	$173,326			$58,867	$8,126	$31,467	$16,720	$15,248	$212	$18,337	$20,796	$3,563

Brewery Creek Preliminary Economic Assessment

Base case													
Estimate of Income					>> Begin production								End <<
Description	**Units**	**Total**	**2015**	**2016**	**2017**	**2016**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**
Cash Flow													
Operating Margin	$000s	$173,326	$	$	$58,857	$8,126	$31,467	$16,720	$15,248	$212	$18,337	$20,798	$3,563
Project Capital	$000s	-$89,410	-$5,770	-$76,770	-$2,710	$	$	-$4,161	$	$	$	$	$
Mine Closure Costs	$000s	-$8,000											-$8,000
Royalty (Yukon and Private)	$000s	-$16,342	$	$	-$3,501	-$1,125	-$2,559	-$1,863	-$1,750	-$914	-$1,925	-$2,302	-$401
Income Tax (Federal and Yukon)	$000s	-$14,864	$	$	$	$	-$3,155	-$2,308	-$2,243	$	-$2,394	-$4,423	-$340
Working Capital (Required and adjusted)	$000s	$	$	$	-$8,839	$4,247	$21	$26	$267	-$252	$4	$1,216	$3,311
Pre-Tax and Royalty Cash Flow	**$000s**	**$77,043**	**-$5,770**	**-$76,770**	**$47,309**	**$12,373**	**$31,488**	**$12,585**	**$15,515**	**-$40**	**$18,342**	**$22,011**	**-$1,127**
Cumulative	$000s	**$77,043**	**-$5,770**	**-$82,539**	**-$35,230**	**-$22,858**	**$8,630**	**$21,216**	**$36,730**	**$36,690**	**$56,032**	**$77,043**	**$75,916**
Net Present Value in US$000s	5%	**$45,658**											
IRR	%	**22%**											
Payback	**months**	**33**					4						
"	**years**	**2.7**											
Post-Tax Cash Flow	**$000s**	**$46,578**	**-$5,770**	**-$76,770**	**$43,808**	**$11,247**	**$26,774**	**$8,414**	**$11,622**	**-$956**	**$14,022**	**$16,286**	**-$1,868**
Cumulative	$000s	**$46,578**	**-$5,770**	**-$82,539**	**-$38,732**	**-$27,484**	**-$1,710**	**$6,704**	**$18,225**	**$17,271**	**$31,293**	**$46,578**	
Net Present Value in US$000s	5%	**$23,315**											
IRR	%	**15%**											
Payback	**months**	**38**						4					
"	**years**	**3.2**											
Peak Funding	**$000s**	**-$76,770**											
PROJECT CAPITAL													
Project Capital													
Equipment & Structures			Initial	Initial	Sustaining	Sustaining	Sustaining	Sustaining	Sustaining	Sustaining	Sustaining	Sustaining	Sustaining
10. Overall site	$000s	$67	$	$67	$	$	$	$	$	$	$	$	$
20. Open pit Mining	$000s	$1,679	$42	$1,637	$	$	$		$	$	$	$	$
30. Ore handling	$000s	$17,975	$	$17,975	$	$	$	$	$	$	$	$	$
40. Process	$000s	$40,498	$	$35,425	$946	$		$4,128					
50. Water management	$000s	$99	$	$99	$								
60. Environmental	$000s	$	$	$	$								
70. On site infrastructure	$000s	$3,286	$286	$2,707	$293	$	$	$	$				
Exploration Expenses													
25. Capitalised mining costs	$000s	$11,370	$239	$11,131									
Indirect	$000s	$8,438	$3,877	$3,056	$1,471	$	$	$33	$	$	$	$	$
Owners costs (G and A year -2 and -1)	$000s	$5,998	$1,325	$4,674									
Total Capital	$	**$89,410**	**$5,770**	**$76,770**	**$2,710**	$	$	**$4,161**	$	$	$	$	$
ESTIMATE OF TAX													
Income Tax													
NSR	$000s	**$463,089**	$	$	$94,524	$45,011	$68,261	$53,258	$49,641	$36,576	$54,727	$47,437	$13,654
Freight & Marketing	$000s	**$1,117**	$	$	$228	$109	$165	$128	$120	$88	$132	$114	$33
Operating Costs	$000s	**$288,232**	$	$	$35,354	$36,736	$36,568	$36,361	$34,229	$36,244	$36,209	$26,484	$10,046
Operating Profit	**$000s**	**$173,740**	$	$	$58,942	$8,166	$31,528	$16,768	$15,293	$244	$18,386	$20,838	$3,575
Canadian Exploration Expense (CEE)	$000s	**-$50,108**			-$50,108	$	$	$	$	$	$	$	$
Capital Cost Allowance (CCA)	$000s	**-$64,622**	$	$	-$14,559	-$11,258	-$8,782	-$6,597	-$5,460	-$4,615	-$3,462	-$2,596	-$1,947
Net Income	**$000s**	**$59,010**	$	$	**-$5,726**	**-$3,092**	**$22,746**	**$10,181**	**$9,832**	**-$4,371**	**$14,926**	**$18,242**	**$1,628**
Loss Carry-Forward													
Additions	$000s		$	$	$5,726	$3,092	$	$	$	$4,371	$	$	$
Opening Balance	$000s		0	$	$5,726	$8,818	$8,818	$	$	$4,371	$4,371	$	$
Losses Used	$000s		0	$	$	$	$8,818	$	$	$	$4,371	$	$
Closing Balance	$000s		0	$	$5,726	$8,818	$	$	$	$4,371	$	$	$
Loss Carry-Forward	**$000s**		$	$	**$5,726**	**$3,092**	**-$8,818**	$	$	**$4,371**	**-$4,371**	$	$
Taxable Income													
Net Income	$000s	$59,010	$	$	-$5,726	-$3,092	$22,746	$10,181	$9,832	-$4,371	$14,925	$18,242	$1,628
Loss-Carry-Forward	$000s	-$23,020	$	$	$	$	-$8,818	$	$	$	-$4,371	$	$
Royalty	$000s	-$16,342	$	$	-$3,501	-$1,125	-$2,559	-$1,863	-$1,750	-$914	-$1,925	-$2,302	-$401
Yukon Taxable Income	**$000s**	**$19,646**	$	$	**-$9,227**	**-$4,217**	**$11,369**	**$8,318**	**$8,082**	**-$5,286**	**$8,626**	**$15,940**	**$1,227**
Yukon Territory Income Tax	15%	$8,035	$	$	$	$	$1,705	$1,248	$1,212	$	$1,294	$2,391	$184
Federal Taxable Income	**$000s**	**$11,613**	$	$	**-$9,227**	**-$4,217**	**$9,663**	**$7,070**	**$6,870**	**-$5,286**	**$7,334**	**$13,649**	**$1,043**
Federal Tax	15%	$6,829	$	$	$	$	$1,450	$1,061	$1,030	$	$1,100	$2,032	$156
Royalty Calculation													
Income for Yukon Royalty	**$000s**	**$36,619**	$	$	**-$5,726**	**-$8,816**	**$13,928**	**$10,181**	**$9,832**	**-$4,371**	**$10,553**	**$16,242**	**$1,628**
Yukon Royalty 1	$000s	**$178**	$	$	$	$	$30	$30	$30	$	$30	$30	$30
Yukon Royalty 2	$000s	**$1,031**	$	$	$	$	$200	$200	$200	$	$200	$200	$31
Yukon Royalty 3	$000s	**$1,490**	$	$	$	$	$300	$300	$290	$	$300	$300	$
Yukon Royalty 4	$000s	**$676**	$	$	$	$	$275	$13	$	$	$39	$350	$
Yukon Royalty 5	$000s	**$259**	$	$	$	$	$	$	$	$	$	$259	$
Yukon Royalty 6	$000s	**$**	$	$	$	$	$	$	$	$	$	$	$
Yukon Royalty 7	$000s	**$**	$	$	$	$	$	$	$	$	$	$	$
Yukon Royalty 8	$000s	**$**	$	$	$	$	$	$	$	$	$	$	$
Yukon Royalty 9	$000s	**$**	$	$	$	$	$	$	$	$	$	$	$
Till Capital Royalty at 0.5% NSR	$000s	**$2,316**	$	$	$473	$225	$341	$266	$248	$183	$274	$237	$68
Alexco Royalty at 2% NSR	$000s	**$9,189**	$	$	$1,890	$900	$1,349	$1,054	$982	$732	$1,083	$926	$272
Franco Nevada Royalty at $40/oz. on 21,516 oz.	$000s	**$861**			$861								
Energold Royalty at 5% of Net Profits for Kokanee, Fosters, Golden	$000s	**$342**			$277	$	$64	$	$	$	$	$	$
Percentage of Profits applied to Energold based on ounces mined	%	**2.21**			0.31	0.47	0.29	0.93	0.21				
Total Royalty	9%	**$16,342**	$	$	**$3,601**	**$1,126**	**$2,669**	**$1,863**	**$1,760**	**$914**	**$1,926**	**$2,302**	**$401**

23.0 ADJACENT PROPERTIES

Although numerous quartz claims exist adjacent the Brewery Creek Property no significant mining projects are known to exist. First Nations land of the Tr'ondek Hwech'in exist to the western and southern portions of the property, details of any historical agreements with the first nation are discussed in Section 4.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

Prior to project execution, a project execution plan is required. This will ensure that procurement, construction and planning for the construction period are coherent thereby avoiding costly scheduling delays. Of particular importance is that planning takes into consideration that much of the construction may be restricted to non-winter periods. Additionally provision should be made for health and safety and environmental impacts throughout project construction and operation periods. Tetra Tech EBA has detailed provisions for health safety and environmental concerns and a preliminary schedule is provided for the construction period of the project.

24.1.1 Health, Safety, Environmental and Security

Health, safety and environmental (HSE) programs and initiatives are essential to project success. A fully-integrated program will be implemented to help achieve a “zero-harm” goal. To achieve this, key project stakeholders will be asked to share in this responsibility by providing the leadership and commitment to attain the highest standards and values.

A high level of communication, motivation, and involvement will be required in the development of HSE practices, including alignment with site contractors on topics such as safety training, occupational health and hygiene, hazard and risk awareness, safe systems of work, and job safety analysis. Tools will be implemented for performance tracking and accountability, including procedures for incident management. The Project will incorporate HSE parameters as key criteria in the design, constructability, and operability of each facility and major area.

All design and engineering stages incorporate criteria for responsible management of process flows, effluent and waste products to meet established capture and containment guidelines. The design also incorporates basic clean plant design standards, including operational safety and maintenance access requirements. A hazard and operability analysis (HAZOP) will be conducted by the Project design team during the detailed design stage for each area of the plant. This analysis will strive to eliminate identified hazards. This systematic team approach will identify hazards associated with operability that requires attention in order to eliminate undesirable consequences.

Environmental protection will be incorporated in both the design of the main processes of the plant, as well as in the transportation, storage and disposal of materials within and outside the boundaries of the plant.

24.2 Project Scheduling

The overall Project schedule (the Schedule) identifies the preferred sequence and target milestone dates that should be managed for the Project to be executed successfully. The Schedule assumes completion of Bankable Feasibility work and a project decision document prior commencement of a conceptual two year pre-production detailed design and construction schedule, referred to as year -2 and year -1 in the mine plan. Year 1 represents the first year of mining and production. The Bankable Feasibility work is excluded from the Schedule.

The start dates presented in the Schedule are based on the likely date of reception of the decision document allowing commencement with mining and are subject to change; however the approximate duration and sequence of activities is considered reasonable for this report.

The Schedule is based on a construction season of May 1 through October 31, which is typical for mining projects in the Yukon. It is currently assumed that no building construction, heap leach construction, bulk earthworks or concrete work will be conducted over the winter months, as this would involve increased unit costs. On the basis that building erection will take place in the summer months, construction activities such as installation of processing equipment, indoor electrical distribution systems and indoor structural work is assumed to be possible during winter months. Activities completed during winter months will be conducted under the discretion of the construction manager.

The short duration of the construction season dictates that the execution of the tasks in the Schedule is critical to limiting the project construction period to 12 calendar months, prior to the first gold doré pour. The schedule drawn up for the PEA indicates that the critical path is the engineering, procurement and subsequent construction of the processing equipment and processing facility along with the construction and commissioning of the new heap leach facility.

A list of project milestones is provided in Table 24-1.

Table 24-1: Milestones Time Line

Milestone	Expected duration	Target completion date
Permitting and bankable feasibility	1 Year	3 months prior to start of year -2
Appointment of EP Consultants and Construction Manager		February, year -2
Design engineering of heap leach, crushing and ADR facility	6 months	August, year -2
Procurement of long lead time equipment	9 months	July, year -1
Appointment of earthworks contractor		May, year -2
Temporary construction camp	1 month	May, year -2
Open pit clear and grubbing	1 month	End year -2
Prestripping of waste	6 months	End of year -1
Prestripping leach pad area cells 8-10	1 month	May, year -1
Construction of cells 8-10	5 months	October, year -1
Construction of significant equipment	6 months	End of year -1
Crusher and initial stacking commissioning	1 Month	End of year -1
Plant commissioning	1 month	January, year 1
Commence Mining		May, year 1
First dore production	30 days	June, year 1

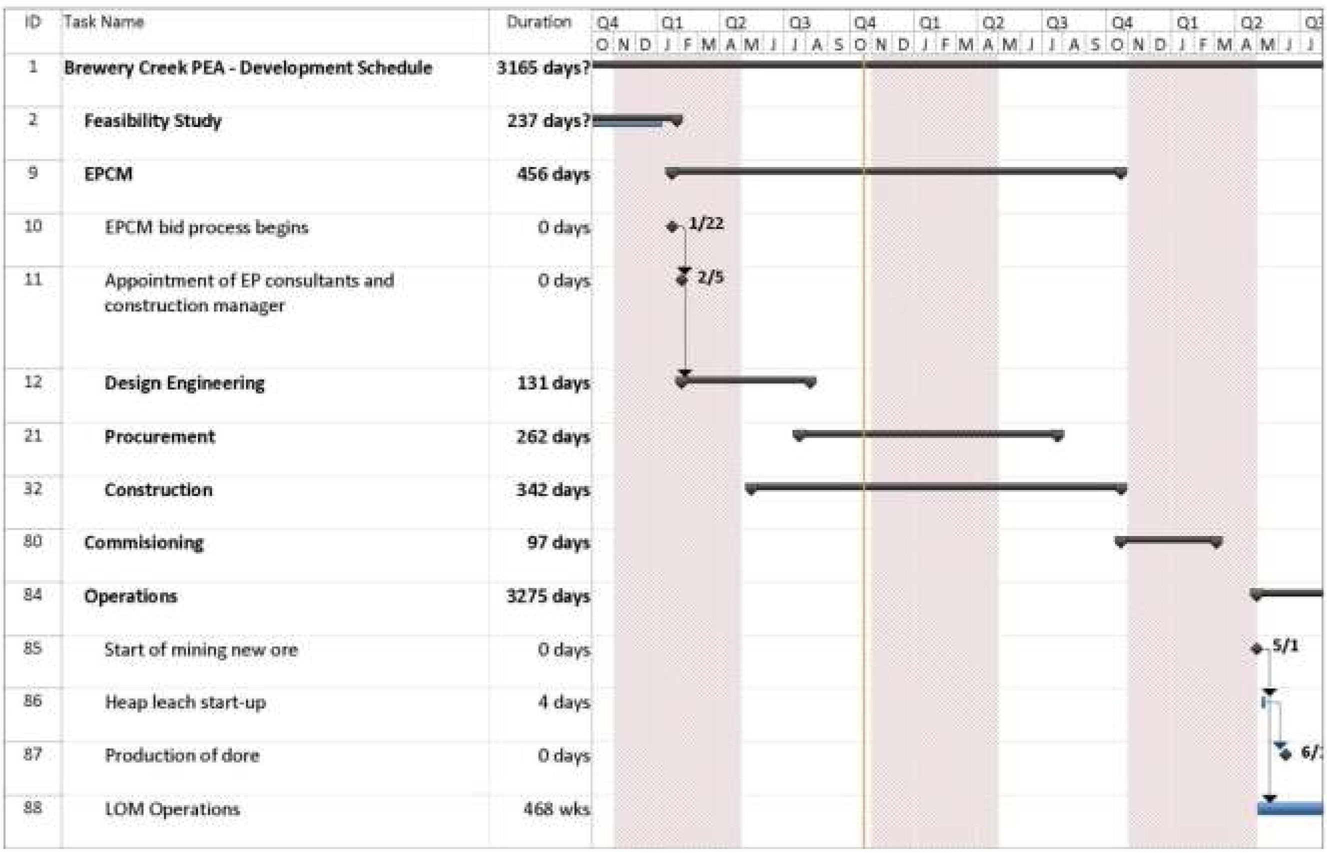
Engineering deliverables and activities will be incorporated into the construction schedule from information pertaining only to the “Issued for Tender” and “Issued for Construction” drawings. These elements constitute the basic needs for development of the various contract scopes and the forms of tender.

Procurement activities related to the need for capital equipment will be derived from the engineer’s equipment list as well as bulk materials, such as piping, electrical cable, lighting, structural steel and concrete, that may be purchased by the Owner. Equipment and bulk materials expediting will be undertaken by the construction management personnel, while the engineers will expedite the capital equipment shop drawings needed to complete design work.

The initiation of contract packages will be based on the timing of the engineering deliverables and the stated deliveries of the capital equipment and bulk materials. Ongoing project scheduling will involve input from all project groups, including the Owner’s operating personnel, consultants and contractors as the Project progresses towards start-up. Individual contractor schedules will be based on the overall schedule, and each contractor is expected to provide a two or three-week look-ahead schedule that will provide a short-term list of activities to be accomplished in that time frame. In addition, the look-ahead schedule will be used to identify potential conflicts with other contractor schedules, resource availability, or other potential issues in an effort to maintain the critical-path for the overall schedule.

The following schedule displays a general projected timeline for achieving the project milestones.

Figure 24–1: Projected Brewery Creek Development Schedule



25.0 INTERPRETATION AND CONCLUSIONS

25.1 General (Tetra Tech EBA)

25.1.1 Risks

Geology and Resource

Several of the mineral resource areas are characterized by mineralization hosted in relatively thin intrusive sills that are in immediate contact with highly preg robbing carbonaceous sediments. Mining these zones will be challenging in order to exclude preg robbing material from being hauled to the leach facility. Depending upon the degree of practical mining selectivity, a portion of the resource may be lost due to mining constraints. Golden Predator will need to by highly cognizant of this issue as the deposits are developed.

A re-processing cut-off grade of 0.30 gpt Au has been applied to the resources stated for the historical heap leach material. Grade control in this highly heterogeneous material may be challenging and will require development of a reliable procedure to be implemented.

Economics

Project economics is based on revenues that are derived from indicated and inferred resources, and actual costs could vary from the estimated costs, resulting in a change in the economics of the project. In addition the assumption of a consistent gold price throughout the project is not realizable if the project is developed. The economic analysis only provides an indication in current funds for the project viability for use in project comparison and for decision making on whether or not to proceed with further feasibility studies for the project.

Significant project revenue is lost to royalties. Golden Predator management may wish to review these terms and opt to exercise some of the buyout opportunities.

Construction Materials

Construction materials are planned to be locally derived. Inventory of local quarries and scheduling of suitable pit waste material has been taken into account in the mine plan. Limited test work has been conducted to characterize the source material to ensure it is sufficient in quantity and suitable for the proposed purposes, including impermeable sub-liner material for the heap leach cells 8-10 construction, foundation materials for stockpile and crusher foundations, and road resurfacing.

Scheduling

It has been assumed that a 2 year pre-production period is required for the project development. This excludes the time required to complete a bankable feasibility study as well as permitting. With respect to the northern climate associated with this project, the open air construction season is limited. Some construction may be possible within the site buildings outside of the construction season. Plans for engineering design and procurement need to take this into account.

25.1.2 Opportunities

Geology and Resources

The Brewery Creek area contains numerous shallow gold deposits. To date, no high-grade feeder zones or large mineralized masses have been identified in the project area. Drilling in the Lone Star area in 2012, in addition to results from Classic, resulted in the recognition of a potential bulk tonnage deposit with a local higher-grade zone of skarn-style mineralization for consideration in project expansion. Gold grades for the currently identified deposits tend to be relatively low so the potential for higher grades in the Lone Star area could help future project economics.

Project size

Based on the results of this PEA, there may be an opportunity to evaluate lower process throughput options. This could have a positive influence on project economics by reducing capital costs and lengthening the life of mine.

Mining

The weathered and softer overburden depth for the project is not well understood. Deposit modelling which identifies a layer of overburden, which may not require blasting may help reduce mining costs estimates. If further geotechnical work can find ways to produce steeper pit slopes safely, some of the deposits could be mined with lower stripping ratios and deepening of deposits may be possible.

Economics

Due to the project location and the climatic conditions, the PEA considers shut down of 1/3 of each operating year. There may be opportunity in extending the mining and crushing season of pits or the old heap leach facility. This additional material would be processed the following year and could act as insulation during the cold season.

The additional tonnage mined each year is expected to result in lower costs per tonne for mining and processing, as it is expected fixed cost components will remain the same as current. Should this be found feasible, the entire project could be re-evaluated on lower unit costs and potentially more tonnage over life of mine.

25.2 General (SGS)

25.2.1 Risks

Operations in sub-arctic conditions require special considerations in terms of operational efficiency of personnel and equipment. The Brewery Creek Property, as proposed, operates as a part-time process, ceasing mining and active crushing during the cold part of the season and assumes an operating season commencing in March and ending in late October. Crushing and conveying equipment will be idle for several months and exposed to extreme temperatures. Some personnel will stay on during the off season to monitor and operate solution recirculating systems.

The project depends on Laura Creek to provide makeup water to the process plant during the operating season. Laura Creek water may not be available when operations are scheduled to begin due to a long freeze or harsh weather conditions.

25.2.2 Opportunities

The previous operation installed; but never operated; a treatment plant for the removal of contaminants from heap leach solutions for discharge to Laura Creek. A similar facility is proposed for the current design. It may be possible to delay the construction of the treatment plant or refine the requirements and use a less expensive option once the project is in operation.

25.3 Truck Stacking

25.3.1 Risks

The Brewery Creek leach pad will be stacked using 40-tonne trucks. The trucks will impart additional compaction to the crushed ore pad. This compaction is a common problem with leach pads which causes ponding, internal hydrostatic pressure build-up in buried lifts, and blinding. The most common practice used to relieve compaction is to cross-rip the leach pad using a track-dozer towing a long ripper shank after materials have been placed. Ripping can destroy the agglomerates if performed too soon after stacking before the materials have had time to cure.

The effect of compaction from truck stacking on the Brewery Creek materials is unknown at this time. Permeability test-work does indicate that the agglomerated materials are permeable up to 60 meters at the application rate proposed; tests also indicate the settlement of materials ranges from 26 to 42 percent on three columns tested. This settlement may pose physical and practical challenges to truck stacking. Ramps created to access the pad tend to be the areas where compaction issues generally occur because they receive heavy traffic for longer periods that the cells areas.

25.3.2 Opportunities

Truck stacking can be more flexible than conveyor stacking and may allow the operator to increase production when the crushing plant output exceeds nominal production, place materials on the pad during daylight hours only, or reduce the hours of placement to a single, 12-hour shift.

25.4 Plant Site

25.4.1 Opportunities

Crushing Plant Site Preparation and Earthwork

The crushing plant pad elevation may be revised or reconfigured as an opportunity to reduce earthwork and associated cost. Further geotechnical investigations are warranted to confirm and finalize the design and volumes.

Crushing Plant Alternatives

The proposed crushing plant layout is a proposal based on currently available quotations. Additional layouts from other vendors maybe available to reduce costs and this alternative should be explored in the next phase of engineering.

Mobile equipment could be considered for initial start-up if replacement later with modular equipment is capital justified. Mobile equipment differs from modular equipment in its expected operational life because it is lighter. A reasonable expectation of use for mobile equipment is three years.

Process Plant Site Preparation and Earthwork

An existing concrete slab remains from the previous operation process plant. Further investigation of the conditions of the slab is recommended in order to determine if it can be remain as base, or if it can be re-used as part of part of the process plant slab, to avoid removal, and its associated cost.

25.5 Metallurgical Testwork

25.5.1 Risks

The Brewery Creek Property has historically mined areas where preg-robbing materials have been identified. Some of those materials may have been loaded on to the old heap leach. Crushing old leach to produce materials with an 80% passing size of 9 mm will expose the operation to some risk unless a program is defined to reduce the amount of potential preg-robbing.

26.0 RECOMMENDATIONS

26.1 Recommendations

Tetra Tech EBA on the basis of this PEA recommended the following priorities to enhance viability with respect to the Brewery Creek Property:

1. Complete remaining infill drilling on all deposits included in the PEA to increase confidence of Inferred Resources to an Indicated level, and conduct confirmatory drilling at Kokanee, Golden and Lucky to validate historical results for these areas ($1M),

2. Continue with metallurgical and processing test work for the existing deposits and initiate test work for the Classic and Lonestar deposits, initiate metallurgical test work at Kokanee, Golden and Lucky ($500k),
3. Continue with Executive Committee Project Proposal document and associated site investigation and surveys along with the development of a Mine Closure and Reclamation Plan ($750k),
4. Review the potential success of Fort Knox, Alaska operation as a year around operation. This operation is located in similar climate and has just started to test year around mining and leaching. If possible, this may have a significant positive impact on project economics,
5. Conduct a trade-off study to test effect of various production rates on initial capital requirements,
6. Commence assessment of Classic and Lonestar deposits, and increase confidence of current resource; conduct trade-offs to assess viability of these areas as integrated or a stand-alone operations on the property.

An estimated budget of $2.25M is anticipated to be required to fulfil items 1 through 3. Budgets have not been estimated for items 4 through 6 as these may be conducted as internal exercises and may have indirect value added to the current PEA.

The above recommendations should be attempted before a Feasibility level study and associated detailed site work is considered.

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CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE

I, Mark Horan do hereby certify that:

1. I am senior mining engineer at Tetra Tech EBA Inc. at 1066 West Hastings Street, Vancouver, British Columbia, Canada V6E 3X2;
2. I am a graduate of the University of Witwatersrand, 1997, with a BSc. Mining Engineering and I am a graduate of Rhodes University, 2002 with an MSc.;
3. Since 1998 to present I have been employed in the mining industry in various roles, I have worked in gold, coal, chrome and industrial mineral mining. I have been author of technical reports for mining operations in South Africa, Mexico and Canada;
4. I am a Registered Professional Engineer, with the Association of Professional Engineers and Geoscientists of British Columbia), registration number 170768;
5. I am a "Qualified Person" for the purposes of National Instrument 43-101;
6. I am responsible for the preparation of the technical report titled “Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada”, effective date July 22, 2014 and dated [November 19, 2014] (the “Technical Report”), with specific responsibility for Sections 1, 15, 16, 18.1, 18.2, 18.6 through 18.9, 18.10.1, 18.10.2, 18.10.4, 18.10.6, 18.11 through 18.13, 18.14.2, 18.14.3, 18.14.4, 18.14.7, 18.14.8, 18.15, 18.16.5, 18.18, 18.19, 19, 21.1.1 through 21.1.4, 21.2.1 through 21.2.4, 24, 25.1, 26 and 27.
7. My most recent personal inspection of the property was on October 16-18, 2012;
8. I am independent, as described in Section 1.5 of National Instrument 43-101, of Golden Predator Corp., the corporation for which I prepared portions of the Technical Report;
9. I have read National Instrument 43-101 and this Technical Report has been prepared in compliance with National Instrument 43-101; and
10. As of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed (*Mark Horan*)

“ Mark Horan ”

____

Mark Horan
Senior Mining Engineer
Tetra Tech EBA Inc.

November 19, 2014

CERTIFICATE

I, James Barr HEREBY CERTIFY THAT:

1. I am Team Lead and a senior geologist at Tetra Tech EBA Inc. at 1066 West Hastings, Vancouver, British Columbia, V6E 3X2;
2. I am a registered Professional Geoscientist with the Association of Professional Engineers and Geoscientists of the province of British Columbia (#35150);
3. From 2003 to the present I have worked as an exploration and resource geologist for numerous gold projects in northern Canada and Mexico, and have worked on a precious metal oxide heap leach project in Mexico;
4. I visited the project between March 19-21, 2012, and most recently between May 30-31, 2012;
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101;
6. I was the project manager and responsible for the preparation of Sections 2 through 10, 11.1, 11.2, 11.3, 11.5.3, 12.1, 12.2, 14.4, 14.5.4, 20 and 23 of the technical report titled “Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada”, effective date July 22, 2014 and dated November 19, 2014.
7. I am independent, as described in Section 1.5 of National Instrument 43-101, of Golden Predator Corp., the corporation for which I prepared portions of the Technical Report;
8. I was co-author of the recent report titled “NI 43-101 Technical Report on Resources, Brewery Creek Project, Yukon Territory, Canada,” dated October 23rd, 2013 effective date June 1st, 2013; and was principal author of the previous Brewery Creek Technical Report titled “Updated Mineral Resource Estimate for the Brewery Creek Property, Yukon Territory, Canada, prepared for Golden Predator Canada Corp” with Effective Date of March 11, 2012 and amended date of January 17, 2013;
9. I have read National Instrument 43-101 and this Technical Report has been prepared in compliance with National Instrument 43-101; and
10. As of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed

“ *James Barr* “

James Barr
Senior Geologist
Tetra Tech EBA Inc.

November 19, 2014

CERTIFICATE

I, Marvin Silva HEREBY CERTIFY THAT:

1. I am a Senior Geotechnical Engineer with Tetra Tech Inc. with a business address at 2015 West River Road, Tucson, Arizona 85704;
2. I am a graduate of the National Autonomous University of Nicaragua (B.Sc. Agricultural Engineering, 1981); Institute of Odessa in Ukraine (M.Sc. Water Resources Engineering, 1985); and University of Alberta (PhD Geoenvironmental Engineering, 1999).
3. My relevant experience includes 22 years of experience in geotechnical engineering, including 8 years in the mining industry.
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of Alberta (#52477).
5. I am a "Qualified Person" for the purposes of National Instrument 43-101;
6. I am responsible for the preparation of the technical report titled "Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada", effective date July 22, 2014 and dated November 19, 2014 (the "Technical Report"), with specific responsibility for Sections 17.1 and 21.1.12.
7. I am independent, as described in Section 1.5 of National Instrument 43-101, of Golden Predator Corp., the corporation for which I prepared portions of the Technical Report;
8. I have not had any prior involvement with the property prior to the heap leach design work conducted for this report;
9. I have read National Instrument 43-101 and this Technical Report has been prepared in compliance with National Instrument 43-101; and
10. As of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed (*Marvin Silva*)

" Marvin Silva "

____

Marvin Silva
Senior Geotechnical Engineer
Tetra Tech, Inc.

November 19, 2014

CERTIFICATE

I, Nick Michael, BS, MBA, HEREBY CERTIFY THAT:

1. I am a Principal Mineral Economist with Tetra Tech, Inc. with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401, USA.

2. I am a graduate of the Colorado School of Mines in Golden, Colorado USA in mining engineering (1983) and received and received an MBA from Willamette University (1986).

3. I have practiced my profession continuously since 1987. Since 1990, I have completed valuations, evaluations (technical-economic models), and have audited a variety of projects including exploration, pre-production (feasibility-level), operating and mine closure projects. I have also served as expert witness with respect to technical-economic issues.

4. I am a Registered Member in good standing (#4104304) with the Society for Mining, Metallurgy and Exploration Inc. (SME).

5. I have read the definition of “qualified person” set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.

6. I am responsible for the preparation of the technical report titled “Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada”, effective date July 22, 2014 and dated November 19, 2014 (the “Technical Report”), with specific responsibility for Section 22.

7. As a Qualified Person for this report, I have read the National Instrument 43-101 and Companion Policy and confirm that this report has been prepared in compliance to National Instrument 43-101.

8. I have never visited nor had involvement with the Brewery Creek property prior to this report.

9. I am independent of Golden Predator Exploration Inc. as independence is described in Section 1.5 of the National Instrument 43-101. In addition I am currently not a shareholder of Golden Predator Exploration Inc. nor am I directly entitled to financially benefit from its success.

10. Prior to this report, I have had no prior involvement on this property.

11. To the best of my knowledge, information and belief, as of the effective date of the Technical Report, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed (*Nick Michael*)

" Nick Michael "

Nick Michael
Senior Minerals Economist
Tetra Tech Inc.
November 19, 2014

CERTIFICATE

I, Donald E. Hulse, P.E., SME-RM HEREBY CERTIFY THAT:

1. I am currently employed as Vice President - Mining by Gustavson Associates, LLC at:

   274 Union Boulevard
   Suite 450
   Lakewood, Colorado 80228

2. I am a graduate of the Colorado School of Mines with a Bachelor of Science in Mining Engineering (1982), and have practiced my profession continuously since 1983.

3. I am a registered Professional Engineer in the State of Colorado (35269), and a registered member of the Society of Mining Metallurgy & Exploration (1533190RM).

4. I have worked as a mining engineer for a total of 30 years since my graduation from university; as an employee of a major mining company, a major engineering company, and as a consulting engineer. I have performed resource estimation and mine planning on numerous gold and silver deposits for over 11 mining companies in six countries working as a consultant as well as an engineer or engineering manager with direct production responsibility for the projects.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am currently responsible for the preparation of the technical report entitled "Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada", with effective date July 22, 2014, and dated November 19, 2014 (the "Technical Report"), with specific responsibility for Sections 14.1, 14.5.1 and 14.6.

7. I was responsible for the preparation of the previous technical report entitled "NI 43-101 Technical Report on Resources, Brewery Creek Project, Yukon Territory, Canada", effective date June 1st, 2013, and dated October 23rd, 2013 (the "Technical Report"), with specific responsibility for Sections 14.1,14.5 and 14.6.

8. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101, and the Technical Report has been prepared in compliance with that instrument and form.

10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed (*Donald E. Hulse*)

" Donald E. Hulse "

Donald E. Hulse
Principal Mining Engineer
Gustavson Associates, LLC

November 19, 2014

CERTIFICATE

I, M. Claiborne Newton, III, Ph.D., SME-RM HEREBY CERTIFY THAT:

1. I am currently employed as Associate Chief Geologist: Gustavson Associates, LLC

   274 Union Boulevard
   Suite 450
   Lakewood, Colorado 80228

2. I am a graduate of North Carolina State University with a Bachelor of Arts in Geology (1977), a Master of Science degree in Geological Sciences (1983) from Virginia Polytechnic Institute and State University and a Doctorate of Philosophy in Geosciences (1990) from the University of Arizona. I have practiced my profession continuously since 1977.

3. I am a Registered Member in good standing of the Society for Mining, Metallurgy and Exploration (SME, #4145342RM) a Qualified Professional Member in good standing of the Mining and Metallurgical Society of America (MMSA, #01396QP) with recognized special expertise in geology, mining, and ore reserves, and a registered Professional Geologist in the State of Virginia (#2801001736). I am also a member of the Society of Economic Geologists.

4. I have worked as a geologist for a total of 37 years since graduation from university - as an employee of three major mining companies and two major engineering and geological consulting firms, as a consulting geologist and as a university instructor.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of the technical report titled "Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada", effective date July 22, 2014 and dated November 19, 2014 (the "Technical Report"), with specific responsibility for Sections 11.4.2, 11.5.2, and 12.4. I most recently visited the property for 2 days on June 4th and 5th, 2013.

7. I was co-author and responsible for Sections 1 through 12.3, 15 through 20, and the overall content of the technical report titled "NI 43-101 Technical Report on Resources, Brewery Creek Project, Yukon Territory, Canada," dated October 23rd, 2013 effective date June 1st, 2013.

8. I am independent of the issuer, applying all of the tests in Section 1.5 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101, and the Technical Report has been prepared in compliance with that instrument and form.

10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed (M. *Claiborne Newton*)

" M. Claiborne Newton "

M. Claiborne Newton
Associate Chief Geologist
Gustavson Associates, LLC

November 19, 2014

CERTIFICATE of AUTHOR

I, Joseph M. Keane, P.E. do hereby certify that:

1. I am an Independent Mineral Processing Engineer Consultant and contributed to a Report entitled "Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada", effective date July 22, 2014 and dated November 19, 2014 as an associate of the following organization:

SGS E&S Engineering Solutions
7701 N. Business Park Drive
Tucson, Arizona 85743
Telephone: 520-579-8315
Fax: 520-579-3686
E-Mail: Joseph.Keane@sgs.com

2. This certificate applies to the Report titled "Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada", effective date July 22, 2014 and dated November 19, 2014 (the "Technical Report").

3. I graduated with a degree of Bachelor of Science in Metallurgical Engineering from the Montana School of Mines in 1962. I obtained a Master of Science in Mineral Processing Engineering in 1966 from the Montana College of Mineral Science and Technology. In 1989 I received a Distinguished Alumni Award from that institution.

4. I am a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME #1682600) and the Instituto de Ingenieros de Minas de Chile. I am a registered professional metallurgical engineer in Arizona (#12979) and Nevada #5462).

5. I have worked as a metallurgical engineer for a total of 52 years since my graduation from university.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7. I am responsible for Sections 13, 17.2, 18.3 through 18.5, 18.10.3, 18.10.5, 18.14.1, 18.14.5, 18.14.6, 18.16.1 through 18.16.4, 18.17, 21.1.5, through 21.1.11, 21.2.5, 25.2 through 25.5. I visited the property in October 2012.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. As of the date of this certificate, to the best of my knowledge, information, and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites assessable by the public.

Signed and Sealed

"Joseph M. Keane"

Joseph M. Keane, P.E.
Principal Metallurgical Engineer
SGS - E&S Engineering Solutions Inc.

November 19, 2014

CERTIFICATE

I, Michael J. Lechner, C.P.G. HEREBY CERTIFY THAT:

1. I am a consulting geologist and President of Resource Modeling Incorporated (RMI), an Arizona corporation with a business address of 124 Lazy J Drive, PO Box 295, Stites, ID 83552;

2. I am a graduate of the University of Montana with a B.A. degree in Geology (1979);

3. From 1979 to the present I have been actively employed in various capacities of the mining industry. I have worked as an exploration geologist exploring for precious and base metals throughout western North America (8 years), a mine geologist working at precious metal mines in California and Nevada (10 years), and a geologic consultant during which time I have estimated Mineral Resources for numerous precious and base metal deposits located throughout the world (17 years);

4. I am a Registered Professional Geologist in the State of Arizona (#37753), a Certified Professional Geologist with the American Institute of Professional Geologists (#10690), a P. Geo. with British Columbia (#155344), and a Registered Member of SME (#4124987RM);

5. I am a "Qualified Person" for the purposes of National Instrument 43-101;

6. I am responsible for the preparation of the technical report titled “Preliminary Economic Assessment for the Brewery Creek Property, Yukon Territory, Canada”, effective date July 22, 2014 and dated November 19, 2014 (the “Technical Report”), with specific responsibility for Sections 11.4.1, 11.5.1, 12.3, 14.2, 14.3, 14.5.2, 14.5.3.

7. I was co-author and responsible for Sections 12.2,14.2, 14.3, 14.5, and 14.6 of the technical report titled “NI 43-101 Technical Report on Resources, Brewery Creek Project, Yukon Territory, Canada,” dated October 23rd, 2013, with effective date June 1st, 2013.

8. My most recent personal inspection of the property was on October 16-18, 2012;

9. I am independent, as described in Section 1.5 of National Instrument 43-101, of Golden Predator Corp., the corporation for which I prepared portions of the Technical Report;

10. I have read National Instrument 43-101 and this Technical Report has been prepared in compliance with National Instrument 43-101; and

11. As of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed (*Michael J. Lechner*)

" Michael J. Lechner "

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Michael J. Lechner
President
Resource Modeling Inc.
November 19, 2014