MINE DEVELOPMENT ASSOCIATES
A Division of RESPEC

RESPEC

TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE DELAMAR AND FLORIDA MOUNTAIN GOLD - SILVER PROJECT, OWYHEE COUNTY, IDAHO, USA

Submitted to:

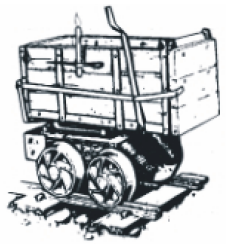
INTEGRA RESOURCES
1050-400 Burrard Street
Vancouver, BC, Canada V6C 3A6
www.integraresources.com

Date of Report: October 22, 2019

Effective Date: September 9,2019

Qualified Persons:
Michael M. Gustin, C.P.G.
Steven I. Weiss, C.P.G.
Thomas L. Dyer, P.E.
Jack S. McPartland, Member M.M.S.A.
Jeffrey L. Woods, Member S.M.E., M.M.S.A.
John D. Welsh, P.E.

775-856-5700
210 S Rock Blvd
Reno, NV 89502
www.mda.com



RESPEC

MINE DEVELOPMENT ASSOCIATES

A Division of **RESPEC**

CONTENTS

1.0 SUMMARY 1
- 1.1 Property Description and Ownership 1
- 1.2 Exploration and Mining History 2
- 1.3 Geology and Mineralization 3
- 1.4 Drilling, Database and Data Verification 4
- 1.5 Metallurgical Testing 5
- 1.6 Estimated Mineral Resources 6
- 1.7 Mining Methods 10
- 1.8 Processing and Recovery Methods 11
- 1.9 Capital and Operating Costs 12
- 1.10 Preliminary Economic Analysis 14
- 1.11 Conclusions and Recommendations 15

2.0 INTRODUCTION AND TERMS OF REFERENCE 16
- 2.1 Project Scope and Terms of Reference 16
- 2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure 17

3.0 RELIANCE ON OTHER EXPERTS 20

4.0 PROPERTY DESCRIPTION AND LOCATION 21
- 4.1 Location 21
- 4.2 Land Area 22
- 4.3 Agreements and Encumbrances 24
- 4.4 Environmental Liabilities and Permitting 26

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY 28
- 5.1 Access to Property 28
- 5.2 Physiography 29
- 5.3 Climate 29
- 5.4 Local Resources and Infrastructure 29

6.0 HISTORY 30
- 6.1 Carson Mining District Discovery and Early Mining: 1863 – 1942 30
- 6.2 Historical Exploration Since the 1960s 32
- 6.3 Modern Historical Mining: 1977 through 1998 34
- 6.4 Historical Resource and Reserve Estimations 36

775-856-5700

210 South Rock Blvd.
Reno, Nevada 89502
FAX: 775-856-6053

7.0 GEOLOGIC SETTING AND MINERALIZATION 39
7.1 Regional Geologic Setting 39
7.2 Owyhee Mountains and District Geology 40
7.3 DeLamar Project Area Geology 41
7.3.1 DeLamar Area 41
7.3.2 Florida Mountain – Stone Cabin Mine Area 45
7.4 Mineralization 50
7.4.1 District Mineralization 50
7.5 DeLamar Project Mineralization 51
7.5.1 DeLamar Area 51
7.5.1.1 Milestone Prospect 55
7.5.2 Florida Mountain Area 55

8.0 DEPOSIT TYPE 58

9.0 EXPLORATION 60
9.1 Topographic and Geophysical Surveys 60
9.2 Rock and Soil Geochemical Sampling 61
9.3 Database Development and Checking 61
9.4 Cross-Sectional Geologic Model 62

10.0 DRILLING 63
10.1 Summary 63
10.2 Historical Drilling – DeLamar Area 64
10.2.1 Continental 1966 64
10.2.2 Earth Resources 1969 - 1970 64
10.2.3 Sidney Mining 1972 66
10.2.4 Earth Resources ~1970 – 1983 66
10.2.5 NERCO 1985 - 1992 67
10.2.6 Kinross 1993 - 1998 67
10.3 Historical Drilling – Florida Mountain Area 67
10.3.1 Earth Resources 1972 – 1976 67
10.3.2 ASARCO 1977 68
10.3.3 Earth Resources 1980 68
10.3.4 NERCO 1985 – 1990 68
10.3.5 Kinross 1995 – 1997 68
10.4 Integra Drilling 2018 - 2019 69
10.4.1 DeLamar Area Drilling 2018 - 2019 69
10.4.2 Florida Mountain Area Drilling 2018 70
10.5 Drill-Hole Collar Surveys 70
10.6 Down-Hole Surveys 70
10.7 Sample Quality and Down-Hole Contamination 71
10.8 Summary Statement 71

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY 73
11.1 Historical Sample Preparation and Security 73
11.2 Integra Sample Handling and Security 73
11.3 Historical Sample Analysis – Prior to Commercial Open-Pit Mining Operations 74
11.4 Historical Sample Analysis – During Commercial Open-Pit Mining Operations 74
11.5 Integra Sample Analysis 75
11.6 Quality Assurance / Quality Control Programs 76
11.6.1 Historical Operators 76
11.6.2 Integra 77
11.7 Summary Statement 77

12.0 DATA VERIFICATION 78
12.1 Drill-Hole Data Verification 78
12.1.1 Collar and Down-Hole Survey Data 78
12.1.2 Assay Data 79
12.1.3 Integra Data Verification 79
12.2 Quality Assurance/Quality Control Results 80
12.2.1 Historical QA/QC Results 80
12.2.2 Integra QA/QC Results 84
12.3 Additional Data Verification 89
12.4 Site Inspection 90
12.5 Independent Verification of Mineralization 90
12.6 Summary Statement 91

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING 92
13.1 DeLamar Area Production 1977 – 1992 92
13.1.1 DeLamar Area Mill Production 1977 - 1992 92
13.1.2 Cyanide Heap Leaching 1987 - 1990 93
13.2 Mineralogy from Metallurgical Studies 94
13.3 Historical Metallurgical Testing 95
13.3.1 1970s Earth Resources – Hazen Testwork 96
13.3.2 1989 Sullivan Gulch Testing for NERCO 97
13.3.3 1980s Florida Mountain Testing for NERCO 98
13.4 Integra 2018-2019 Metallurgical Tests 100
13.4.1 DeLamar Area Testing 2018-2019 101
13.4.1.1 DeLamar Heap Leach Testing 103
13.4.1.2 DeLamar Agitated Cyanide Leach Testing 109
13.4.1.3 DeLamar Gravity Concentration and Flotation Testing 112
13.4.2 Florida Mountain Area Testing 114
13.4.2.1 Florida Mountain Comminution Testing 115
13.4.2.2 Florida Mountain Heap Leach Testing 115
13.4.2.3 Florida Mountain Agitated Cyanide Leach Testing 119
13.4.2.4 Florida Mountain Gravity Concentration and Treatment of Gravity Tailings 119
13.4.2.5 Florida Mountain Flotation Concentrate Regrind/Agitated Leach 121
13.5 Summary Statement 123

14.0 MINERAL RESOURCE ESTIMATES 125
14.1 Introduction 125
14.2 DeLamar Project Data 127
14.2.1 Drill-Hole Data 128
14.2.2 Topography 128
14.2.3 Modeling of Historical Underground Workings 129
14.3 Geological Modeling 129
14.4 Deposit Geology Pertinent to Resource Modeling 129
14.5 Water Table 130
14.6 Oxidation Modeling 130
14.7 Density Modeling 131
14.8 DeLamar Area Gold and Silver Modeling 132
14.8.1 Mineral Domains 132
14.8.2 Assay Coding, Capping, and Compositing 140
14.8.3 Block Model Coding 142
14.8.4 Grade Interpolation 143
14.8.5 Model Checks 144
14.9 Florida Mountain Area Gold and Silver Modeling 144
14.9.1 Mineral Domains 145
14.9.2 Assay Coding, Capping, and Compositing 146
14.9.3 Block Model Coding 152
14.9.4 Grade Interpolation 152
14.9.5 Model Checks 153
14.10 DeLamar Project Mineral Resources 154
14.11 Discussion of Resource Modeling 169

15.0 MINERAL RESERVE ESTIMATES 171

16.0 MINING METHODS 172
16.1 Economic Parameters 172
16.2 Cutoff Grades 173
16.3 Geometric Parameters 175
16.4 Pit Optimization 175
16.4.1 DeLamar Pit Optimization 176
16.4.2 Florida Mountain Pit Optimization 178
16.5 Road and Ramp Design 182
16.6 Pit Design 182
16.7 In-Pit Resources 183
16.8 Dump and Leach Pad Design 185
16.9 Mine Production Schedule 186
16.10 Equipment Requirements 193
16.11 Personnel Requirements 195

17.0 RECOVERY METHODS 197
17.1 Summary Process Design Criteria 197
17.2 Process Descriptions 199
17.2.1 Phase I: Florida Mountain 30 ktpd Heap Leach 199
17.2.2 Phase II: Florida Mountain 2 ktpd Concentrator with Leach Process Description 201
17.2.3 Phase III: DeLamar 30 ktpd Heap Leach Process Description 204
17.3 Process Water, Energy and Materials 204

18.0 PROJECT INFRASTRUCTURE 206
18.1 Heap-Leach Pad Construction 206
18.2 Dry Stack Tailings Construction 207
18.3 Mine Site Personnel 207

19.0 MARKET STUDIES AND CONTRACTS 209
19.1 Metal Pricing 209

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT 210
20.1 Environmental Baseline Studies 210
20.2 Permitting 211
20.2.1 Environmental Impact Statement 211
20.2.2 Idaho Pollutant Discharge Elimination System Permit ("IPDES") 212
20.2.3 U.S. Army Corps of Engineers Section 404 Dredge and Fill Permit 212
20.2.4 Biological Opinion 212
20.2.5 Major State Authorizations, Licenses, and Permits 213
20.2.6 Local County Requirements 215
20.2.7 Idaho Joint Review Process 215
20.2.8 EIS / Permitting Timelines and Costs 215
20.2.9 Most Likely Case EIS Cost Summary 217
20.2.10 Integra Permitting Management Strategy 218
20.2.10.1 Permitting Risks and Risk Management Strategy 218
20.3 Social and Community 219
20.4 Waste Characterization 219
20.5 Closure and Reclamation Strategy 219

21.0 CAPITAL AND OPERATING COSTS 221
21.1 Mining Capital 222
21.1.1 Primary Equipment 223
21.1.2 Support Equipment 223
21.1.3 Blasting Equipment 223
21.1.4 Mine Maintenance Capital 224
21.1.5 Other Capital 224
21.1.6 Mine Preproduction Costs 224
21.2 Process Capital 224
21.2.1 Heap-Leach Pad Capital 226
21.2.2 Tailings Impoundment 227

21.3 Owner and Other Capital Costs....227
21.3.1 Light Vehicles....228
21.3.2 Preproduction Owner Costs....230
21.4 Reclamation Costs and Salvage Value....230
21.5 Mine Operating Costs....230
21.5.1 Mine General Services....231
21.5.2 Mine Maintenance....232
21.5.3 Drilling....233
21.5.4 Blasting....234
21.5.5 Loading....235
21.5.6 Hauling....236
21.5.7 Mine Support....237
21.6 Process Operating Cost Summary....237
21.7 G&A Costs....241

22.0 ECONOMIC ANALYSIS....243
22.1 Mining Physicals....244
22.2 Pre-Tax Cash Flow....245
22.3 Tax Considerations & After-Tax Cash Flow....246
22.4 Sensitivity Analyses....247

23.0 ADJACENT PROPERTIES....251

23.0 OTHER RELEVANT DATA AND INFORMATION....252

24.0 INTERPRETATION AND CONCLUSIONS....253
24.1 DeLamar Project Opportunities....255
24.2 DeLamar Project Risks....257

25.0 RECOMMENDATIONS....258

26.0 REFERENCES....260

27.0 DATE AND SIGNATURE PAGE....265

28.0 CERTIFICATE OF QUALIFIED PERSONS....266

TABLES

Table 1.1 Summary of DeLamar Area Grade Estimation Parameters....7
Table 1.2 Pit Optimization Cost Parameters....8
Table 1.3 Pit-Optimization Metal Recoveries by Deposit and Oxidation State....8
Table 1.4 Total DeLamar Project Gold and Silver Resources....9
Table 1.5 DeLamar Area Gold and Silver Resources....9
Table 1.6 Florida Mountain Area Gold and Silver Resources....9
Table 1.7 Capital Cost Summary....13
Table 1.8 Operating and Total Cost Summary....14

Table 1.9 Preliminary Economic Analysis Summary 14
Table 1.10 Summary of Integra Estimated Costs for Recommended Program 15
Table 4.1 Summary of Estimated Land Holding Costs for the DeLamar Project 24
Table 4.2 Summary of Agreements and Encumbrances 25
Table 6.1 DeLamar Mine Gold and Silver Production 1977 – 1992 35
Table 6.2 Historical Resource and Reserve Estimates 37
Table 7.1 Summary of Volcanic Rock Units in the Vicinity of the DeLamar Mine 42
Table 10.1 DeLamar Project Drilling Summary 63
Table 10.2 Historical Drilling at the DeLamar and Florida Mountain Areas 64
Table 10.3 Integra Drilling Summary 69
Table 12.1 Integra Certified Reference Materials 85
Table 13.1 1987 – 1990 Heap Leach Summary 93
Table 13.2 Historical Mineralogy and Metallurgical Testing, DeLamar and Florida Mountain Deposits 96
Table 13.3 Summary Results, Rougher Flotation Tests, Hazen 1970 Studies 96
Table 13.4 Summary Results, Agitated Cyanidation Tests, Hazen 1970 Studies 97
Table 13.5 Summary Results of Hazen Testing, Sullivan Gulch Sample 98
Table 13.6 NERCO Florida Mountain Column-Leach Tests 99
Table 13.7 Other NERCO Florida Mountain Column-Leach Tests 99
Table 13.8 Drill Hole Composite Summary, DeLamar 2018-2019 Testing 102
Table 13.9 Summary 2018-2019 DeLamar Bottle-Roll Test Results, 80%-1.7mm Feed Size 104
Table 13.10 Summary 2018-2019 Sullivan Gulch Bottle-Roll Tests, 80%-1.7mm Feed Size 105
Table 13.11 DeLamar 2018-2019 Summary Column-Leach Cyanidation Tests on Bulk Samples 107
Table 13.12 Grind Agitated Bottle Roll Tests, DeLamar 2018-2019 Drill-Core Composites 110
Table 13.13 Gravity Concentration with Flotation of Gravity Rougher Tailings 113
Table 13.14 Summary of Flotation Test Results and Optimization Testing, 2018-2019 114
Table 13.15 Florida Mountain 2018-2019 Drill Hole Composite Summary 115
Table 13.16 Florida Mountain 2018-2019 Bottle-Roll Test Results 116
Table 13.17 Florida Mountain 2018-2019 Column-Leach Results 117
Table 13.18 Florida Mountain 2018-2019 Whole Ore" Milling & Cyanidation Tests 119
Table 13.19 Florida Mountain 2018-2019 Gravity Concentration - Agitated Cyanidation of Gravity Tailings 120
Table 13.20 Florida Mountain 2018-2019 Gravity Concentration with Flotation of Gravity Tailings 120
Table 13.21 Florida Mountain 2018-2019 Gravity Concentration, Flotation of Gravity Tailings, Regrind Leach of Flotation Concentrate 121
Table 13.22 DeLamar Project 2019 PEA Recovery and Reagent Estimates 124
Table 14.1 Integra Specific Gravity Determinations from DeLamar Deposit Drill Core 131
Table 14.2 Integra Specific Gravity Determinations from Florida Mountain Deposit Drill Core 132
Table 14.3 Approximate Grade Ranges of DeLamar Area Gold and Silver Domains 132
Table 14.4 DeLamar Area Gold and Silver Assay Caps by Domain 140
Table 14.5 Descriptive Statistics of DeLamar Area Coded Gold Assays 141
Table 14.6 Descriptive Statistics of DeLamar Area Coded Silver Assays 141
Table 14.7 Descriptive Statistics of DeLamar Area Gold Composites 142
Table 14.8 Descriptive Statistics of DeLamar Area Silver Composites 142
Table 14.9 Summary of DeLamar Area Grade Estimation Parameters 143
Table 14.10 Approximate Grade Ranges of Florida Mountain Area Gold and Silver Domains 145

Table 14.11 Florida Mountain Area Gold and Silver Assay Caps by Domain 146
Table 14.12 Descriptive Statistics of Florida Mountain Area Coded Gold Assays 151
Table 14.13 Descriptive Statistics of Florida Mountain Area Coded Silver Assays 151
Table 14.14 Descriptive Statistics of Florida Mountain Area Gold Composites 151
Table 14.15 Descriptive Statistics of Florida Mountain Area Silver Composites 152
Table 14.16 Summary of Florida Mountain Area Estimation Parameters 153
Table 14.17 Pit Optimization Cost Parameters 154
Table 14.18 Pit-Optimization Metal Recoveries by Deposit and Oxidation State 154
Table 14.19 Total DeLamar Project Gold and Silver Resources 155
Table 14.20 DeLamar Area Gold and Silver Resources 155
Table 14.21 Florida Mountain Area Gold and Silver Resources 156
Table 14.22 Resource Classification Parameters 156
Table 14.23 Total Project In-Pit Oxidized and Transitional Mineralization at Various Cutoffs 168
Table 14.24 Total Project In-Pit Unoxidized Mineralization at Various Cutoffs 169
Table 16.1 DeLamar and Florida Mountain Economic Parameters 173
Table 16.2 DeLamar and Florida Mountain Recoveries 173
Table 16.3 DeLamar and Florida Mountain AuEq Factors 174
Table 16.4 DeLamar and Florida Mountain Cutoff Grades (g Au/t) 175
Table 16.5 DeLamar Pit Optimization Results 177
Table 16.6 DeLamar Pit by Pit Results 177
Table 16.7 Florida Mountain Pit Optimization Results 179
Table 16.8 Florida Mountain Pit by Pit Results 180
Table 16.9 DeLamar In-Pit Resources 183
Table 16.10 Florida Mountain In-Pit Resources 184
Table 16.11 Total PEA In-Pit Resources 185
Table 16.12 Waste and Leach Containment Requirements 186
Table 16.13 DeLamar Mine Production Schedule 188
Table 16.14 Florida Mountain Mine Production Schedule 189
Table 16.15 Total PEA Mine Production Schedule 190
Table 16.16 PEA Process Production Schedule 191
Table 16.17 Florida Mountain Leach Stockpile Balance 192
Table 16.18 DeLamar Leach Stockpile Balance 192
Table 16.19 Florida Mountain Mill Stockpile Balance 193
Table 16.20 PEA Yearly Mine Equipment Requirements 193
Table 16.21 Schedule Efficiency 194
Table 16.22 PEA Mining Personnel Requirements 196
Table 17.1 DeLamar Process Development Phase 197
Table 17.2 DeLamar Project Process Design Criteria 197
Table 18.1 Mine, Process and Administrative Personnel 207
Table 21.1 Capital Cost Summary 221
Table 21.2 Operating and Total Cost Summary 222
Table 21.3 Mining Capital Cost by Year 223
Table 21.4 Yearly Heap Leach Processing Capital Costs 225
Table 21.5 Yearly Florida Mountain Concentrator Processing Capital Costs 226
Table 21.6 Infrastructure & Owners Capital 228
Table 21.7 Light Vehicle Cost Estimate 229

Table 21.8 Yearly Mine Operating Cost Estimate 231
Table 21.9 Mine General Services, Engineering and Geology Costs 232
Table 21.10 Yearly Mine Maintenance Costs 233
Table 21.11 Yearly Drilling Costs 234
Table 21.12 Yearly Blasting Costs 235
Table 21.13 Yearly Loading Costs 236
Table 21.14 Yearly Haulage Costs 237
Table 21.15 Yearly Mine Support Costs 237
Table 21.16 Florida Mountain Heap Leach - Power Costs 238
Table 21.17 Florida Mountain Concentrator - Power Costs 238
Table 21.18 DeLamar Heap Leach - Power Costs 238
Table 21.19 Florida Mountain Heap Leach Consumables Costs 239
Table 21.20 Florida Mountain Concentrator with Concentrate Leach Consumables Costs 239
Table 21.21 DeLamar Heap Leach Consumables Costs 239
Table 21.22 Florida Mountain Heap Leach OPEX Summary 240
Table 21.23 Florida Mountain Concentrator OPEX Summary 240
Table 21.24 DeLamar Heap Leach OPEX Summary 240
Table 21.25 Yearly Process Operating Costs 241
Table 21.26 Yearly G&A Costs 242
Table 22.1 Economic Analysis Summary 243
Table 22.2 Yearly Mine & Process Physicals 245
Table 22.3 Pre-Tax Cash Flow 246
Table 22.4 Depreciation, Depletion, Taxes, and After-Tax Cash Flow 247
Table 22.5 Project Sensitivity to Metal Prices 248
Table 22.6 Revenue Sensitivity (After Tax) 248
Table 22.7 Operating Cost Sensitivity (After Tax) 248
Table 22.8 Capital Cost Sensitivity (After Tax) 249
Table 22.9 Gold Recovery Sensitivity 250
Table 22.10 Silver Recovery Sensitivity 250
Table 25.1 Integra Cost Estimate for the Recommended Program 259

FIGURES

Figure 4.1 Location Map, DeLamar Gold – Silver Project 21
Figure 4.2 Property Map for the DeLamar Project 23
Figure 5.1 Access Map for the DeLamar Project 28
Figure 6.1 Estimated Annual Production Value, Silver City (Carson) Mining District 1863-1942 31
Figure 6.2 Aerial View, Zones of Exploration and Mining Since 1969 within the DeLamar Area 33
Figure 6.3 Aerial View of the Florida Mountain (Stone Cabin Mine) Area 35
Figure 6.4 Photograph of the Reclaimed Florida Mountain (Stone Cabin) Mine Area 36
Figure 7.1 Shade Relief Map with Regional Setting of the Owyhee Mountains 39
Figure 7.2 Geologic Map of the Central Owyhee Mountains 40
Figure 7.3 Land Position Map Showing Mineralized Zones 43
Figure 7.4 Integra Generalized 2018 DeLamar Area Geology 44
Figure 7.5 Integra 2018 Schematic Cross-Section, DeLamar Area 44
Figure 7.6 Volcano-Tectonic Setting of the DeLamar Area 45

Figure 7.7 Geologic Map of Florida Mountain 47
Figure 7.8 Map Legend for Florida Mountain Geology 48
Figure 7.9 Schematic Florida Mountain Cross Section (Looking Northeast) 49
Figure 7.10 Veins of the Historical De Lamar Mine, Elevation 6,240 Feet 52
Figure 7.11 Longitudinal Section of the Black Jack – Trade Dollar Mine 57
Figure 8.1 Schematic Model of a Low-Sulfidation Epithermal Mineralizing System 58
Figure 9.1 Plan View of Resistivity from 2017 and 2018 IP/RES Surveys 60
Figure 9.2 Plan View of Chargeability from 2017 and 2018 IP/RES Surveys 61
Figure 10.1 Map of DeLamar Area Drill Holes 65
Figure 10.2 Map of Florida Mountain Area Drill Holes 66
Figure 12.1 Repeat Mine Lab Silver Assays Relative to Original Mine Lab Assays 81
Figure 12.2 Outside Lab Silver Assays Relative to Original Mine Lab Assays 82
Figure 12.3 Mine Lab Silver AA Analyses Relative to Mine Lab Silver Fire Assays 83
Figure 12.4 Mine Lab Gold AA Analyses Relative to Mine Lab Gold Fire Assays 83
Figure 12.5 CRM CDN-GS-P6A Gold Analyses 86
Figure 12.6 CRM SN74 Silver Analyses 87
Figure 12.7 Coarse Blank Gold Values vs. Gold Values of Previous Samples 88
Figure 12.8 RC Field Duplicate Gold Results Relative to Primary Sample Assays 89
Figure 13.1 DeLamar Area 2018-2019 Composites: CN/FA vs. Sulfide Sulfur (%) 103
Figure 13.2 DeLamar 2018-2019 Composites Bottle-Roll Tests, Gold Recovery vs. Sulfide Sulfur 106
Figure 13.3 Gold Leach Rate Profiles for Column-Leach Tests, DeLamar Bulk Samples, ... 107
Figure 13.4 Gold Recovery in Column-Leach Tests vs. Bottle Roll Tests, DeLamar Bulk Samples 108
Figure 13.5 Gold Recovery, Unoxidized Sample Grind-Leach, 2018-2019 Drill Core Composites 111
Figure 13.6 Silver Recovery, Unoxidized Sample Grind-Leach, 2018-2019 Drill Core Composites 111
Figure 13.7 Florida Mountain 2018-2019 Gold Leach Rate Profiles for Column-Leach Tests 118
Figure 13.8 Florida Mountain 2018-2019 Gold and Silver Leach Rates, Agitated Cyanidation of Flotation Rougher Concentrate 122
Figure 14.1 Cross Section 1230 NW Showing Sullivan Gulch Gold Domains 134
Figure 14.2 Cross Section 1230 NW Showing Sullivan Gulch Silver Domains 135
Figure 14.3 Cross Section 2010 NW Showing Sommercamp and N. DeLamar Gold Domains 136
Figure 14.4 Cross Section 2010 NW Showing Sommercamp and N. DeLamar Silver Domains 137
Figure 14.5 Cross Section 2790 NW Showing Gold Domains at Glen Silver 138
Figure 14.6 Cross Section 2790 NW Showing Silver Domains at Glen Silver 139
Figure 14.7 Florida Mountain Cross Section 2830 N Showing Geology and Gold Domains 147
Figure 14.8 Florida Mountain Cross Section 2830 N Showing Geology and Silver Domains 148
Figure 14.9 Florida Mountain Cross Section 3280 N Showing Geology and Gold Domains 149
Figure 14.10 Florida Mountain Cross Section 3280 N Showing Geology and Silver Domains 150
Figure 14.11 Cross Section 1230 NW Showing Sullivan Gulch Block-Model Gold Grades 158
Figure 14.12 Cross Section 1230 NW Showing Sullivan Gulch Block-Model Silver Grades 159
Figure 14.13 Cross Section 2010 NW Showing Sommercamp – Regan and N. DeLamar Block-Model Gold Grades 160
Figure 14.14 Cross Section 2010 NW Showing Sommercamp – Regan and N. DeLamar Block-Model Silver Grades 161
Figure 14.15 Cross Section 2790 NW Showing Glen Silver Block-Model Gold Grades 162
Figure 14.16 Cross Section 2790 NW Showing Glen Silver Block-Model Silver Grades 163
Figure 14.17 Cross Section 2830 N Showing Florida Mountain Block-Model Gold Grades 164

Figure 14.18 Cross Section 2830 N Showing Florida Mountain Block-Model Silver Grades 165
Figure 14.19 Cross Section 3280 N Showing Florida Mountain Block-Model Gold Grades..... 166
Figure 14.20 Cross Section 3280 N Showing Florida Mountain Block-Model Silver Grades 167
Figure 16.1 DeLamar Pit by Pit Graph..... 178
Figure 16.2 Florida Mountain Pit by Pit Graph..... 181
Figure 17.1 Florida Mountain 30 ktpd Heap Leach Process Flow Sheet. 200
Figure 17.2 Florida Mountain: 2 ktpd Flotation with Leach 202
Figure 17.3 DeLamar 30 ktpd Heap-Leach Process Flow Sheet..... 205
Figure 18.1 PEA General Arrangement Drawing 208
Figure 22.1 Annual Operating After-Tax Cash Flow..... 244
Figure 22.2 After-Tax Sensitivity..... 249

APPENDICES

Appendix A Listing of Unpatented and Patented Claims and Leased Land
Appendix B Metallurgical Test Results

Frontispiece: view looking northwest to the partly back-filled Sommercamp pit and Sommercamp highwall; top of the north highwall of the Glen Silver pit is barely visible to the left of the trees above the Sommercamp highwall.

1.0 SUMMARY

Mine Development Associates ("MDA") has prepared this technical report and Preliminary Economic Assessment ("PEA") on the DeLamar gold – silver project, located in Owyhee County, Idaho, at the request of Integra Resources Corp. ("Integra"), a Canadian company listed on the TSX Venture Exchange (TSX.V:ITR) and the OTC Markets (OTCQX: IRRZF). The DeLamar project encompasses the DeLamar and Florida Mountain deposit areas. Both deposit areas have been subject to historical underground mines that operated in the late 1800s and early 1900s, as well as late 20th century open-pit mining. The most recent open-pit mining, which ceased in 1998, was conducted by Kinross Gold Corporation ("Kinross").

This report has been prepared under the supervision of Michael M. Gustin, C.P.G. and Senior Geologist for MDA, Thomas L. Dyer, P.E. and Senior Engineer for MDA, Steven I. Weiss, C.P.G. and Senior Associate Geologist for MDA, Jack McPartland, Senior Metallurgist with McClelland Laboratories, Inc., Jeff Woods of Woods Process Services in Denver, Colorado, and John Welsh of Welsh Hagen in Reno, Nevada, in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as amended. Mr. Gustin, Mr. Weiss, Mr. Dyer, Mr. McPartland, Mr. Wood and Mr. Welsh are Qualified Persons under NI 43-101 and have no affiliation with Integra or Kinross except that of independent consultant/client relationships. Mr. Weiss visited the project site on August 1, 2 and 3, 2017, and Mr. Gustin visited the project on October 16, 17, and 18, 2018. Mr. McPartland visited the DeLamar project site on January 17, 2019. Mr. Welsh last visited the property on June 26, 2019.

The effective date of this technical report is September 9, 2019.

1.1 Property Description and Ownership

The DeLamar project consists of 748 unpatented lode, placer, and millsite claims, and 16 tax parcels comprised of patented mining claims, as well as certain leasehold and easement interests, that cover approximately 8,100 hectares in southwestern Idaho, about 80 kilometers southwest of Boise. The property is approximately centered at 43°00′48″N, 116°47′35″W, within portions of the historical Carson (Silver City) mining district, and it includes the formerly producing DeLamar mine last operated by Kinross. The total annual land-holding costs are estimated to be $321,626. All mineral titles and permits are held by the DeLamar Mining Company ("DMC"), an indirect, 100% wholly owned subsidiary of Integra that was acquired from Kinross through a Stock Purchase Agreement in 2017.

775-856-5700

210 South Rock Blvd.
Reno, Nevada 89502
FAX: 775-856-6053

A total of 284 of the unpatented claims were acquired from Kinross, 101 of which are subject to a 2.0% net smelter returns royalty ("NSR") payable to a predecessor owner. This royalty is not applicable to the current project resources.

There are also six lease agreements covering 26 patented claims and one unpatented claim that require NSR payments ranging from 2.5% to 5.0%. One of these leases covers a small portion of the DeLamar area resources and one covers a small portion of the Florida Mountain area resources, with 5.0% and 2.5% NSRs applicable to maximums of $50,000 and $650,000 in royalty payments, respectively.

The property includes 1,355 hectares under six leases from the State of Idaho, which are subject to a 5.0% production royalty of gross receipts plus annual payments of $23,252. One of these leases has been issued and five are pending issuance. The State of Idaho leases include very small portions of both the DeLamar and Florida Mountain resources.

Kinross has retained a 2.5% NSR royalty that applies to those portions of the DeLamar area claims that are unencumbered by the royalties outlined above. The Kinross royalty applies to more than 90% of the current DeLamar area resources, but this royalty will be reduced to 1.0% upon Kinross receiving total royalty payments of CAD$10,000,000.

DMC also owns mining claims and leased lands peripheral to the DeLamar project described above. These landholdings are not part of the DeLamar project, although some of the lands are contiguous with those of the DeLamar and Florida Mountain claims and state leases.

The DeLamar project historical open-pit mine areas have been in closure since 2003. Even though a substantial amount of reclamation and closure work has been completed to date at the site, there remain ongoing water-management activities and monitoring and reporting. A reclamation bond of $2,778,929 remains with the Idaho Department of Lands ("IDL") and a reclamation bond of $100,000 remains with the Idaho Department of Environmental Quality. A reclamation bond in the amount of $51,500 has been placed with the U.S. Bureau of Land Management ("BLM") for exploration activities on public lands.

1.2 Exploration and Mining History

Total production of gold and silver from the DeLamar – Florida Mountain project area is estimated to be approximately 1.3 million ounces of gold and 70 million ounces of silver from 1891 through 1998, with an unknown quantity produced at the DeLamar mill in 1999, and recorded production may have occurred from 1876 to 1891. This includes an estimated 1.025 million ounces of gold and 51 million ounces of silver produced from the original De Lamar underground mine and the later DeLamar open-pit operations. At Florida Mountain, nearly 260,000 ounces of gold and 18 million ounces of silver were produced from the historical underground mines and late 1990s open-pit mining.

Mining activity began in the area of the DeLamar project when placer gold deposits were discovered in 1863 in Jordan Creek, just upstream from what later became the town site of De Lamar. During the summer of 1863, the first silver-gold lodes were discovered in quartz veins at War Eagle Mountain, to the east of Florida Mountain, resulting in the initial settlement of Silver City. Between 1876 and 1888, significant silver-gold veins were discovered and developed in the district, including underground mines at De Lamar Mountain and Florida Mountain. A total of 553,000 ounces of gold and 21.3 million

ounces of silver were reportedly produced from the De Lamar and Florida Mountain underground mines from the late 1800s to early 1900s.

The mines in the district were closed in 1914 and very little production took place until the gold and silver prices increased in the1930s. Placer gold was again recovered from Jordan Creek from 1934 to 1940, and in 1938 a 181 tonne-per-day flotation mill was constructed to process waste dumps from the De Lamar underground mine. The flotation mill reportedly operated until the end of 1942. Including Florida Mountain, the De Lamar – Silver City area is believed to have produced about 1 million ounces of gold and 25 million ounces of silver from 1863 through 1942.

During the late 1960s, the district began to undergo exploration for near-surface bulk-mineable gold-silver deposits, and in 1977 a joint venture operated by Earth Resources Corporation ("Earth Resources") began production from an open-pit milling and cyanide tank-leach operation at De Lamar Mountain, known as the DeLamar mine. In 1981, Earth Resources was acquired by the Mid Atlantic Petroleum Company ("MAPCO"), and in 1984 and 1985 the NERCO Mineral Company ("NERCO") successively acquired the MAPCO interest and the entire joint venture to operate the DeLamar mine with 100% ownership. NERCO was purchased by the Kennecott Copper Corporation ("Kennecott") in 1993. Two months later in 1993, Kennecott sold its 100% interest in the DeLamar mine and property to Kinross, and Kinross operated the mine, which expanded to the Florida Mountain area in 1994. Mining ceased in 1998, milling ceased in 1999, and mine closure activities commenced in 2003. Closure and reclamation were nearly completed by 2014, as the mill and other mine buildings were removed and drainage and cover of the tailings facility were developed.

Total open-pit production from the DeLamar project from 1977 through 1998, including the Florida Mountain operation, is estimated at approximately 750,000 ounces of gold and 47.6 million ounces of silver, with an unknown quantity produced at the DeLamar mill in 1999. From start-up in 1977 through to the end of 1998, open-pit production in the DeLamar area totaled 625,000 ounces of gold and about 45 million ounces of silver. This production came from pits developed at the Glen Silver, Sommercamp – Regan (including North and South Wahl), and North DeLamar areas. In 1993, the DeLamar mine was operating at a mining rate of 27,216 tonnes per day, with a milling capacity of about 3,629 tonnes per day. In 1994, Kinross commenced open-pit mining at Florida Mountain while continuing production from the DeLamar mine. The ore from Florida Mountain, which was mined through 1998, was processed at the DeLamar facilities. Florida Mountain production in 1994 through 1998 totaled 124,500 ounces of gold and 2.6 million ounces of silver.

1.3 Geology and Mineralization

The DeLamar project is situated in the Owyhee Mountains near the east margin of the mid-Miocene Columbia River – Steens flood-basalt province and the west margin of the Snake River Plain. The Owyhee Mountains comprise a major mid-Miocene eruptive center, generally composed of mid-Miocene basalt flows intruded and overlain by mid-Miocene rhyolite dikes, domes, flows and tuffs, developed on an eroded surface of Late Cretaceous granitic rocks.

The DeLamar mine area and mineralized zones are situated within an arcuate, nearly circular array of overlapping porphyritic and flow-banded rhyolite flows and domes that overlie cogenetic, precursor pyroclastic deposits erupted as local tuff rings. Integra interprets the porphyritic and banded rhyolite

flows and latites as composite flow domes and dikes emplaced along regional-scale northwest-trending structures. At Florida Mountain, flow-banded rhyolite flows and domes cut through and overlie a tuff breccia unit that overlies basaltic lava flows and Late Cretaceous granitic rocks.

Gold-silver mineralization occurred as two distinct but related types: (i) relatively continuous, quartz-filled fissure veins that were the focus of late 19th and early 20th century underground mining, hosted mainly in the basalt and granodiorite and to a lesser degree in the overlying felsic volcanic units; and (ii) broader, bulk-mineable zones of closely-spaced quartz veinlets and quartz-cemented hydrothermal breccia veins that are individually continuous for only a few feet laterally and vertically, and of mainly less than 1.3 centimeters in width – predominantly hosted in the rhyolites and latites peripheral to and above the quartz-filled fissures. This second style of mineralization was mined in the open pits of the late 20th century DeLamar and Florida Mountain operations, hosted primarily by the felsic volcanic units.

The fissure veins mainly strike north to northwest and are filled with quartz accompanied by variable amounts of adularia, sericite or clay, ± minor calcite. Vein widths vary from a few centimeters to several meters, but the veins persist laterally and vertically for as much as several hundreds of meters. Principal silver and gold minerals are naumannite, aguilarite, argentite, ruby silver, native gold and electrum, native silver, cerargyrite, and acanthite. Variable amounts of pyrite and marcasite with very minor chalcopyrite, sphalerite, and galena occur in some veins. Gold- and silver-bearing minerals are generally very fine grained.

The gold and silver mineralization at the DeLamar project is best interpreted in the context of the volcanic-hosted, low-sulfidation type of epithermal model. Various vein textures, mineralization, alteration features, and the low contents of base metals in the district are typical of shallow low-sulfidation epithermal deposits worldwide.

1.4 Drilling, Database and Data Verification

As of the effective date of this report, the resource database includes data from 2,718 holes, for a total of 306,078 meters, that were drilled by Integra and various historical operators at the DeLamar and Florida Mountain areas. The historical drilling was completed from 1966 to 1998 and includes 2,625 holes for a total of 275,790 meters of drilling. Most of the historical drilling was done using reverse-circulation ("RC") and conventional rotary methods; a total of 106 historical holes were drilled using diamond-core ("core") methods for a total of 10,845 meters. Approximately 74% of the historical drilling was vertical, including all historical conventional rotary holes.

Integra commenced drilling in 2018. As of the end of April 2019, Integra had drilled a total of 55 RC holes, 36 core holes, and 11 holes commenced with RC and finished with core tails, for a total of 33,573 meters in the DeLamar and Florida Mountain areas combined. All but one of the Integra holes were angled.

The historical portions of the current resource drill-hole databases for the DeLamar and Florida Mountain deposit areas were originally created by MDA using original DeLamar mine digital database files, and this information was subjected to various verification measures by both MDA and Integra. The Integra portion of the drill-hole databases was directly created by MDA using original digital

analytical certificates in the case of the assay tables, or it was checked against original digital records in the case of the collar and down-hole deviation tables. Through these and other verification procedures summarized herein, the authors have verified that the DeLamar data as a whole are acceptable as used in this report.

1.5 Metallurgical Testing

Available results from ongoing metallurgical testing by Integra, at McClelland Laboratories (2018-2019) have been used to select preferred processing methods and estimate recoveries for oxide and transitional material types from both the DeLamar and Florida Mountain deposits, as well as unoxidized (sulfide) material type from the Florida Mountain deposit. Metallurgical testing has also been conducted on unoxidized (sulfide) material from the DeLamar deposit, but that testing has not yet progressed to the level required for processing of that material to be included in the PEA.

Samples used for this 2018-2019 testing, primarily composites of 2018 and 2019 drill core, were selected to represent the various material types contained in the current resources from both the DeLamar and Florida Mountain deposits. Composites were selected to evaluate effects of area, depth, grade, oxidation, lithology, and alteration on metallurgical response. In general, test results indicate that materials from each of the DeLamar and Florida Mountain deposits can most usefully be evaluated by considering the oxidation state (oxidized, transitional, or unoxidized).

Bottle-roll and column-leach cyanidation testing on drill core composites from both the DeLamar and Florida Mountain deposits and on bulk samples from the DeLamar deposit have shown that the oxide and transitional material types from both deposits can be processed by heap-leach cyanidation. Testing on drill core composites from the Florida Mountain deposit has shown that the unoxidized material from that deposit is not amenable to heap leach cyanidation but can be leached using cyanide after grinding. The Florida Mountain unoxidized material also responds well to bulk sulfide flotation treatment, and the resulting flotation concentrate is amenable to agitated cyanide leaching. Highest recoveries from the Florida Mountain unoxidized material were obtained by grinding, followed by gravity concentration and flotation of the gravity tailings, with regrind and agitated cyanidation of the flotation concentrate.

Available metallurgical test results indicate that gold recoveries in the range of 75% to 80%, and silver recoveries of about 30%, can be expected from the DeLamar oxide and transitional material types, by heap leaching at a crush size of 80% -13mm. Agglomeration pretreatment of this material is currently planned, because of the potential for processing of some materials with elevated clay content. Heap leach cyanide consumptions are expected to be reasonably low (about 0.3 – 0.4 kg NaCN/tonne).

In the case of the Florida Mountain oxide and transitional material types, gold recoveries of 85% to 90%, and silver recoveries of about 40%, are expected for heap leaching at an 80% -38mm feed size. Agglomeration pretreatment is not considered to be necessary for these material types. Heap leach cyanide consumptions are expected to be reasonably low (about 0.4 kg NaCN/tonne).

Planned processing of the Florida Mountain unoxidized material type includes grinding, followed by gravity concentration and flotation of the gravity tailings, with regrind and agitated cyanidation of the flotation concentrate. Expected recoveries are about 90% gold and 80% silver. Cyanide consumption

for the concentrate leaching is expected to be equivalent to about 0.2 kg NaCN/tonne, on a mill feed basis.

In the case of the unoxidized material from the DeLamar deposit, 2018-2019 testing has shown that this material type is not amenable to heap-leach cyanidation and is highly variable with respect to response to grinding followed by agitated cyanidation. Reasons for the generally poor and highly variable grind-leach recoveries from this material type are poorly understood at present. Additional testing and mineralogy studies are in progress to gain a better understanding of the observed variability in recoveries. Further testing is also planned to better define what portion of the DeLamar unoxidized material type might be economically processed by simple grind-leach processing. Metallurgical testing has also shown that the DeLamar unoxidized material generally responds well to upgrading by gravity and flotation processing. Testing to evaluate subsequent processing of the resulting concentrate is in progress, but has not been completed as of the effective date of this report. It is expected that flotation concentrate produced from a significant portion of the DeLamar unoxidized materials will not be amenable to agitated leach (cyanidation). It is expected that for these flotation concentrates, some form of oxidative pre-treatment (such as pressure oxidation or roasting) will be required to maximize gold recovery by cyanidation. Alternatively, these concentrates could be shipped off site for toll processing.

1.6 Estimated Mineral Resources

Mineral resources have been estimated for both the Florida Mountain and DeLamar areas of the DeLamar project. The gold and silver resources were modeled and estimated by:

- evaluating the drill data statistically;
- creating low- (domain 100), medium- (domain 200) and high-grade (domain 300) mineral-domain polygons for both gold and silver on sets of cross sections spaced at 30-meter intervals;
- projecting the sectional mineral-domain polygons three-dimensionally to the drill data within each sectional window;
- slicing the three-dimensional mineral-domain polygons along 6-meter-spaced horizontal and vertical planes and using these slices to recreate the gold and silver mineral-domain polygons on level plans and long sections, respectively;
- coding a block model to the gold and silver domains for each of the two deposit areas using the level-plan and long-section mineral-domain polygons;
- analyzing the modeled mineralization geostatistically to aid in the establishment of estimation and classification parameters; and
- using inverse-distance to the third power to interpolate grades into models comprised of 6x6x6-meter blocks using the gold and silver mineral domains to explicitly constrain the grade estimations.

Parameters used in the estimation of gold and silver grades are summarized in Table 1.1.

Table 1.1 Summary of DeLamar Area Grade Estimation Parameters

Estimation Pass – Au + Ag Domain	Search Ranges (meters)			Composite Constraints		
	Major	Semi-Major	Minor	Min	Max	Max/Hole
Pass 1 + 2 - Doman 100	60	60	30	2	12	4
Pass 1 + 2 - Doman 200 + 300 + 0	60	60	30	2	20	4
Pass 3 - Doman 0 + 100 + 200 +300	170	170	170	1	20	4

Restrictions on Search Ranges

Domain	Search Restriction Threshold	Search Restriction Distance	Estimation Pass
Au 100	>0.7 g Au/t	40 meters	1, 2
Au 300	>20 g Au/t	35 meters	1, 2, 3
Ag 300	>400 g Ag/t	35 meters	1, 2, 3
Au 0	>0.5 g Au/t	6 meters	1, 2, 3
Ag 0	>34.29 g Ag/t	9 meters	1, 2, 3

The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded by that domain. The estimated grades for each gold and silver domain coded to a block were then coupled with the partial percentages of the those mineral domains in the block, as well as the outside, dilutionary, domain 0 grades and volumes, to enable the calculation of a single volume-averaged gold and a single volume-averaged silver grade for each block. These single resource block grades, and their associated resource tonnages, are therefore fully block-diluted using this methodology.

The search restrictions place limits on the maximum distances from a block that high-grade population composites can be 'found' and used in the interpolation of gold and/or silver grades into a block. To further avoid the smearing of outlier high grades that are sporadically present in the low-grade gold and silver domains, the maximum number of composites allowed for the estimation in Pass 1 and Pass 2 are less than that used for the higher-grade interpolations.

The gold and silver mineralization commonly exhibits multiple orientations, which led to the use of a number of search orientations to control both the DeLamar and Florida Mountain estimations.

Grade interpolation was completed using length-weighted 3.05-meter (10-foot) composites. The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded to that domain. Blocks coded as having partial percentages of more than one gold and/or silver domain had multiple grade interpolations, one for each domain coded into the block for each metal. The estimated grades for each gold and silver domain coded to a block were coupled with the partial percentages of the those mineral domains in the block, as well as any outside, dilutionary, domain 0 grades and volumes, to enable the calculation of a single volume-averaged gold and a single volume-averaged silver grade for each block. These single final resource block grades, and their associated resource tonnages, are therefore fully block-diluted using this methodology.

The DeLamar project mineral resources have been estimated to reflect potential open-pit extraction and processing by a combination of heap leaching, milling / agitated leaching, and flotation. To meet the requirement of the in-pit resources having reasonable prospects for eventual economic extraction, pit optimizations for the DeLamar and Florida Mountain deposit areas were run using the parameters summarized in Table 1.2 and Table 1.3.

Table 1.2 Pit Optimization Cost Parameters

Parameter	DeLamar	Florida Mtn	Unit
Mining Cost	$ 2.20	$ 2.20	$/tonne mined
Heap Leach Processing	$ 3.35	$ 3.35	$/tonne processed
Mill / Agitated Leach Processing	$	$ 10.00	$/tonne processed
Flotation Processing	$ 12.00	$	$/tonne processed
G&A Cost	$ 4,000	$ 4,000	$1,000s/year
Tonnes per Day	15,000	15,000	tonnes-per-day processed
Tonnes per Year	5,250	5,250	1000s tonnes-per-year processed
G&A per Ton	$ 0.76	$ 0.76	$/tonne processed
Au Price	$ 1,400	$ 1,400	$/oz produced
Ag Price	$ 18	$ 18	$/oz produced
Au Refining Cost	$ 5.00	$ 5.00	$/oz produced
Ag Refining Cost	$ 0.50	$ 0.50	$/oz produced
NSR Royalty	1%	0%	

Table 1.3 Pit-Optimization Metal Recoveries by Deposit and Oxidation State

	DeLamar			Florida Mountain		
Process Type	Oxidized	Transitional	Unoxidized	Oxidized	Transitional	Unoxidized
Leach Recovery - Au	85%	80%	-	85%	80%	-
Leach Recovery - Ag	45%	40%	-	45%	40%	-
Mill/Leach Recovery - Au	-	-	-	-	-	86%
Mill/Leach Recovery - Ag	-	-	-	-	-	63%
Flotation Recovery - Au	-	-	90%	-	-	-
Flotation Recovery - Ag	-	-	95%	-	-	-

The pit shells created using these optimization parameters were applied to constrain the project resources of both the DeLamar and Florida Mountain deposit areas. The in-pit resources were further constrained by the application of a gold-equivalent cutoff of 0.2 g/t to all model blocks lying within the optimized pits that are coded as oxidized or transitional, and 0.3 g/t for blocks coded as unoxidized. Gold equivalency, as used in the application of the resource cutoffs, is a function of metal prices (Table 1.2) and metal recoveries, with the recoveries varying by deposit and oxidation state (Table 1.3).

The total DeLamar project resources, which include the resources for both the DeLamar and Florida Mountain areas, are summarized in Table 1.4. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1.4 Total DeLamar Project Gold and Silver Resources

Classification	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Measured	16,078,000	0.52	270,000	34.3	17,726,000
Indicated	156,287,000	0.42	2,106,000	19.7	98,788,000
Measured + Indicated	172,365,000	0.43	2,376,000	21.0	116,514,000
Inferred	28,266,000	0.38	343,000	13.5	12,240,000

1. Mineral Resources are comprised of all oxidized and transitional model blocks at a 0.2 g AuEq/t cutoff and all unoxidized blocks at a 0.3 g AuEq/t that lie within optimized pits
2. The effective date of the resource estimations is May 1, 2019
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability
4. Rounding may result in apparent discrepancies between tonnes, grade, and contained metal content

The gold and silver resources for the DeLamar and Florida Mountain areas are reported separately in Table 1.5 and Table 1.6, respectively.

Table 1.5 DeLamar Area Gold and Silver Resources

Classification	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Measured	14,481,000	0.51	238,000	36.4	16,942,000
Indicated	105,140,000	0.39	1,334,000	23.4	79,241,000
Measured + Indicated	119,621,000	0.41	1,572,000	25.1	96,183,000
Inferred	21,291,000	0.39	266,000	15.2	10,418,000

1. Mineral Resources are comprised of all oxidized and transitional model blocks at a 0.2 g AuEq/t cutoff and all unoxidized blocks at a 0.3 g AuEq/t that lie within optimized pits
2. The effective date of the DeLamar deposit DeLamar area resources is May 1, 2019
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability
4. Rounding may result in apparent discrepancies between tonnes, grade, and contained metal content

Table 1.6 Florida Mountain Area Gold and Silver Resources

Classification	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Measured	1,597,000	0.63	32,000	15.3	784,000
Indicated	51,147,000	0.47	772,000	11.9	19,547,000
Measured + Indicated	52,744,000	0.47	804,000	12.0	20,331,000
Inferred	6,975,000	0.34	77,000	8.1	1,822,000

1. Mineral Resources are comprised of all oxidized and transitional model blocks at a 0.2 g AuEq/t cutoff and all unoxidized blocks at a 0.3 g AuEq/t that lie within optimized pits
2. The effective date of the Florida Mountain deposit DeLamar area resources is May 1, 2019
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability
4. Rounding may result in apparent discrepancies between tonnes, grade, and contained metal content

1.7 Mining Methods

The PEA considers open-pit mining of the DeLamar and Florida Mountain gold-silver deposits. Note that a PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves. There is no certainty that the economic results of the PEA will be realized.

The methodology used for mine planning to define the economics for the PEA includes definition of economic parameters, pit optimization, creation of pit and waste rock facility designs, creation of production schedules, definition of personnel and equipment requirements, estimation of capital and operating costs, and performance of an economic analysis.

Pit optimization assumed processing of Florida Mountain and DeLamar oxide and transition resources as heap leach, and unoxidized Florida Mountain resources as milled using floatation followed by cyanidation of the concentrates on site. Leach material would be processed at 27,000 tonnes per day and mill material would be processed at 2,000 tonnes per day. Processing of the DeLamar material will require crushing and agglomeration prior to heap leaching.

The resulting pit optimizations were used as the basis for pit designs. The designs used an inner-ramp slope of 45°. DeLamar pit designs utilized five pit phases to establish a mining sequence and Florida Mountain pit designs were completed using three pit phases.

Waste management facility designs were created for the PEA to contain the waste material mined from both the DeLamar and Florida Mountain areas. Some waste material may also be stored in the form of backfill where and when space is available, although this has not been assumed for the PEA and therefore this is a potential opportunity for the project.

Production scheduling was completed with leaching starting with Florida Mountain material and DeLamar leach material being processed starting in year 5 at the same rate as Florida Mountain leach material. Florida Mountain unoxidized material will be stockpiled until the flotation mill is constructed. The start of the 2,000 tonne per day mill will be in year 3 and it will operate at a rate of 720,000 tonnes per year until unoxidized material is exhausted.

The total project mining rate is given a reasonable ramp-up that starts at 2,000 tonnes per day and increases to a life-of-mine maximum of 90,000 tonnes per day in later years.

The PEA has assumed owner mining in order to keep operating costs lower than it would be with contract mining. The production schedule was used along with additional efficiency factors, cycle times, and productivity rates to develop the first-principle hours required for primary mining equipment to achieve the production schedule. Mining anticipates 136-tonne capacity haul trucks loaded by hydraulic shovels. Personnel requirements have been estimated based on the number of people required to operate, supervise, maintain, and plan for operations to achieve the production schedule.

1.8 Processing and Recovery Methods

The PEA envisions the use of two process methods for the recovery of gold and silver:

1) Lower-grade oxide and transition materials from both DeLamar and Florida Mountain will be processed by crushed-ore cyanide heap leaching with stacking on a central heap leach by conveyor, followed by Merrill-Crowe zinc precipitation.
2) Higher-grade material from Florida Mountain will be processed using grinding followed by gravity and flotation concentration, with the concentrates processed by regrinding, agitated-cyanide leaching, counter-current decantation ("CCD"), and Merrill-Crowe zinc precipitation. Flotation tailings will be thickened, filtered, and dry stacked at the tailings storage facility. Concentrate-leach tailings will be added to the heap-leach circuit for further recovery of gold and silver.

Both Florida Mountain and DeLamar oxide and transition ore types have been shown to be amenable to heap-leach processing following crushing. Material will be crushed in two stages to a nominal 100 millimeter size at a rate of 28,000 tonnes per day. Initially, for the Florida Mountain materials, the product of the secondary circuit will be a nominal size of 38 millimeters. Transitioning to DeLamar ore types will require the addition of a tertiary crushing circuit with tertiary screens and cone crushers operating in closed circuit to produce a nominal 13-millimeter product followed by cement agglomeration. Lime will be added to the crushed ore for pH control at a dosage of 1 kilogram/tonne. Cement will be added at 3 kilograms/tonne for agglomeration as required.

Crushed and prepared ore will be transferred to the heap-leach pad using overland conveyors and stacked on the heap using portable or grasshopper conveyors and a radial stacking system. Leach solution will be collected at the base on the heap leach and transferred to the Merrill-Crowe processing plant for recovery of precious metals by zinc precipitation. The zinc precipitate will be filtered, dried, and smelted to produce a precious metal doré product for shipment off site.

Gold and silver recoveries are expected to be 90% and 40%, respectively, for the Florida Mountain oxide heap-leach material. The DeLamar oxide recoveries used in this study are 80% for gold and 30% for silver. Cyanide consumptions for the oxide ore types are 0.4 kilograms/tonne and 0.3 kilograms/tonne for Florida Mountain and DeLamar, respectively.

Transition material gold recoveries are projected to be 85% for Florida Mountain and 75% for DeLamar. Silver recoveries for the transition material are projected to be 40% and 30% for Florida Mountain and DeLamar, respectively. Projected cyanide consumption is 0.4 kilograms per tonne for both the Florida Mountain and DeLamar transition material types.

Higher-grade Florida Mountain unoxidized material will be processed by crushing, grinding, gravity, and flotation concentration, followed by cyanide leaching of the concentrates using CCD and Merrill-Crowe precipitation. This circuit is scheduled to operate at a nominal production rate of 2,000 tonnes per day. For this process, the final crusher product will have a nominal particle size of 6 millimeters and will be fed to the ball mill via two belt feeders at a nominal ore production rate of 88 tonnes per hour. The ball mill discharge will be pumped to a set of two hydrocyclones, one operating and one standby, with the cyclone overflow reporting to the flotation conditioning tank. The cyclone underflow will

report to a centrifugal gravity concentrator. Concentrator reject then reports back to the ball mill for additional grinding. The gravity concentrate will report to the concentrate regrind mill for subsequent processing in the leach circuit.

The flotation feed from the conditioning tank will report to the flotation circuit for sulfide concentration. The flotation concentrate will report to a regrind circuit where it will be ground to a nominal 37 µm before being leached in a conventional leach tank and CCD circuit. The flotation tailings are to be thickened and filtered with the filter cake reporting to the dry stacked tailings storage facility.

Leach solid residue and the pregnant leach solution are separated in the CCD circuit. The pregnant leach solution will report to the heap leach Merrill-Crowe circuit where it will be processed using zinc precipitation for the recovery of gold and silver. The leached residue will be thickened to 60% solids and added to the heap leach material before it is stacked on the heap, thus allowing for additional processing and mitigating the need for a cyanide-rated tailings storage facility.

Recoveries from the Florida Mountain milling/concentrator circuit are expected to be 90% for gold and 80% for silver. Sodium cyanide and lime consumptions are both expected at 0.2 kilograms per tonne of material feed.

1.9 Capital and Operating Costs

Table 1.7 summarizes the estimated life-of-mine ("LOM") capital costs for the project. The LOM total capital costs are estimated as $270.3 million, including $161.0 million in preproduction capital (including working capital) and $109.3 million for sustaining capital (which includes $20.0 million in reclamation costs).

Table 1.8 shows the estimated LOM operating costs for the project, which are estimated to be $7.82 per tonne processed. This includes mining costs which are estimated to be $2.00 per tonne mined. The total cash cost is estimated to be $619 per ounce of gold equivalent and all-in sustaining costs are estimated to be $742 per ounce of gold equivalent.

Table 1.7 Capital Cost Summary

Mine	Pre-Production (1)	Sustaining Yr 1 to Yr 10 (1)	Total LOM (1)
Mining Equipment	$ 32,980	$ 52,014	$ 84,994
Pre-Stripping	$ 7,514	$ -	$ 7,514
Other Mine Capital	$ 6,027	$ 746	$ 6,773
Sub-Total Mine	**$ 46,521**	**$ 52,760**	**$ 99,281**
Processing			
Heap Leach Pad	$ 14,130	$ 19,178	$ 33,308
Heap leach Plant (Incl Crushing and Stacking)	$ 48,449	$ -	$ 48,449
Heap leach: Agglomeration / Crushing (DeLamar Ore)	$ -	$ 20,518	$ 20,518
Florida Mill: Plant	$ -	$ 34,354	$ 34,354
Florida Mill: Dry Stack Tailings	$ -	$ 6,990	$ 6,990
Sub-Total Processing	**$ 62,579**	**$ 81,039**	**$ 143,618**
Infrastructure			
Power	$ 21,714	$ -	$ 21,714
Assay Lab	$ 2,804	$ -	$ 2,804
Other	$ 2,552	$ 974	$ 3,526
Sub-Total Infrastructure	**$ 27,070**	**$ 974**	**$ 28,044**
Owner's Costs	$ 5,819	$ -	$ 5,819
SUB-TOTAL	**$ 141,989**	**$ 134,773**	**$ 276,761**
Other			
Working Capital (2)	$ 13,024	$ (13,024)	$ -
Cash Deposit for Reclamation Bonding (3)	$ 6,000	$ (6,000)	$ -
Salvage Value (4)	$ -	$ (26,426)	$ (26,426)
TOTAL	**$ 161,013**	**$ 89,323**	**$ 250,336**
Reclamation	$ -	$ 20,000	$ 20,000
Total Including Reclamation Costs	**$ 161,013**	**$ 109,323**	**$ 270,336**

(1) Capital costs include contingency and EPCM costs;
(2) Working capital is returned in year 11;
(3) Cash deposit = 30% of bonding requirement. Released once reclamation is completed;
(4) Salvage value for mining equipment and plant; and
(5) Reclamation costs listed here are treated as operating costs in the economic evaluation.

Table 1.8 Operating and Total Cost Summary

	USD / Tonne	
LOM Operating Costs	**Mined**	**Processed**
Mining	$ 2.00	$ 4.18
Processing		$ 3.08
G&A		$ 0.55
Total Site Costs		$ 7.82

LOM Cash Costs and All-in Sustaining Costs	By-Product (1)	Co-Product (2)
Mining	$ 380	$ 317
Processing	$ 280	$ 233
G&A	$ 50	$ 42
Total Site Costs	$ 711	$ 592
Transport & Refining	$ 13	$ 11
Royalties	$ 17	$ 14
Total Cash Costs	$ 741	$ 617
Silver By-Product Credits	$ (272)	$ -
Total Cash Costs Net of Silver by-Product	$ 469	$ 617
Sustaining Capital	$ 131	$ 109
Reclamation	$ 19	$ 16
All-in Sustaining Costs	$ 619	$ 742

(1) By-Product costs are shown as US dollars per gold ounces sold with silver as a credit; and
(2) Co-Product costs are shown as US dollars per gold equivalent ounce.

1.10 Preliminary Economic Analysis

MDA has prepared this PEA for the DeLamar mining project, which includes operations at both the DeLamar and Florida Mountain deposits. A summary of the PEA results is shown in Table 1.9.

Table 1.9 Preliminary Economic Analysis Summary

After-tax NPV (5%)	K USD	$357,572
After-tax NPV (8%)	K USD	$284,448
After-tax NPV (10%)	K USD	$244,454
After-tax IRR	%	43%
After-Tax Payback Period	Years	2.35

Note that a preliminary economic assessment is preliminary in nature and it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves. There is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Some economic highlights include:

- Initial construction period is anticipated to be 18 months;
- After-tax net present value ("NPV") (5%) of $358 million with a 43% after-tax internal rate of return ("IRR") using $1,350 and $16.90 per ounce gold and silver prices, respectively;
- After-tax payback period of 2.35 years;
- Year 2 to 6 gold equivalent production of 148,000 ounces (126,000 oz Au and 1,796,000 oz Ag); and
- Year 1 to 10 gold equivalent average production of 124,000 ounces (103,000 oz Au and 1,660,000 oz Ag);
- After-tax LOM cumulative cash flows of $528 million; and
- Average annual after-tax free cash flow of $61 million once in production.

1.11 Conclusions and Recommendations

The authors conclude that the DeLamar project is a project of merit that warrants significant additional investment. There is an excellent opportunity to expand the extents of the current resources both down dip and along strike; altered and mineralized zones peripheral to the resources warrant additional surface prospection and drilling, and there remains potential for discovering high-grade veins below the current levels of drilling.

A work program with an estimated cost of $14,595,000 is recommended through to the end of 2020, as summarized in Table 1.10. This program includes 20,500 meters of RC and core drilling, further metallurgical testing, geotechnical studies for pit-slope stability and site infrastructure, permitting and environmental expenditures, and the initiation of a preliminary feasibility analysis ("PFS").

Table 1.10 Summary of Integra Estimated Costs for Recommended Program

Item	Estimated Cost US$
Exploration RC Drilling (6,500 meters)	$1,500,000
Infill RC and Core Drilling (6,500 meters)	$2,500,000
Metallurgical / Infill Core Drilling (7,500 meters)	$2,700,000
Geological Mapping, Soil Sampling, Geophysics	$375,000
Land Holding Costs	$320,000
Metallurgy	$1,000,000
Geotechnical Studies	$400,000
Resource Update, Initiate PFS, and Technical Report	$900,000
Permitting	$1,900,000
Care and Maintenance / Reclamation	$2,000,000
Site maintenance and G&A	$1,000,000
Total	$14,595,000

2.0 INTRODUCTION AND TERMS OF REFERENCE

Mine Development Associates ("MDA") has prepared this technical report and Preliminary Economic Assessment ("PEA") on the DeLamar and Florida Mountain gold-silver project ("the DeLamar project"), located in Owyhee County, Idaho, at the request of Integra Resources Corp. ("Integra"), a Canadian company based in Vancouver, British Columbia. Integra entered into a binding stock purchase agreement dated September 18, 2017 with Kinross Gold Corporation ("Kinross") to acquire the Kinross DeLamar Mining Company, then an indirect, wholly owned subsidiary of Kinross, and thereby acquired 100% of its DeLamar gold-silver property. Subsequent to that transaction, Integra has acquired 100% interests in significant additional lands at the adjacent Florida Mountain property, as well as other lands outside of the limits of the project.

Integra is listed on the TSX Venture Exchange (TSX.V: ITR) and the OTC Markets (OTCQX: IRRZF). This report has been prepared in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as amended.

2.1 Project Scope and Terms of Reference

The purpose of this report is to provide a PEA and updated technical summary of the DeLamar gold-silver project. The PEA makes use of the 2019 estimates of mineral resources by Gustin et al. (2019) that have an effective date of May 1, 2019 and remain the current mineral resources for the project.

The DeLamar project lies within the historical Carson (Silver City) mining district of southwestern Idaho. The most recent production from the project occurred in 1977 through 1998 by open-pit mining with both milling and minor cyanide heap-leach processing of gold-silver ores. The mine was placed on care and maintenance in 1999, and later underwent mine closure by Kinross.

In addition to the estimation of the updated DeLamar and Florida Mountain mineral resources, the scope of the work completed by the authors included a review of pertinent technical reports and data provided to the authors by Integra relative to the general setting, geology, project history, exploration and mining activities and results, drilling programs, methodologies, quality assurance, metallurgy, and interpretations. References are cited in the text and listed in Section 20.0.

This report has been prepared under the supervision of Michael M. Gustin, C.P.G. and Senior Geologist for MDA, Thomas L. Dyer, P.E., and Senior Engineer for MDA, Steven I. Weiss, C.P.G. and Senior Associate Geologist for MDA, Jack McPartland, Senior Metallurgist with McClelland Laboratories, Inc., Jeff Woods of Woods Process Services in Denver, Colorado, and John Welsh of Welsh Hagen and Associates in Reno, Nevada. Mr. Gustin, Mr. Weiss, Mr. Dyer, Mr. McPartland, Mr. Wood and Mr. Welsh are Qualified Persons under NI 43-101 and have no affiliation with Integra except that of independent consultant/client relationships.

Mr. Weiss visited the project site on August 1, 2, and 3, 2017, accompanied and assisted by Ms. Kim Richardson of Jordan Valley, Idaho. Ms. Richardson is a geologist who joined the DeLamar mine staff in 1980 and eventually held the positions of Senior Mine Geologist, Mine Superintendent, and Mine Manager before leaving the project in 1997. Mr. Weiss reviewed the property geology, exposures of

mineralized rocks in still accessible open pits, and areas of historical exploration drilling peripheral to the open pits, as well as historical exploration data on file at the DeLamar mine-site office.

Mr. Gustin visited the project site on October 16, 17, and 18, 2018, accompanied by various members of the Integra technical team. Mr. Gustin received updates on the property geology, drilling results to date, and drill-targeting concepts. He also inspected mineralized drill-core intervals from various holes, and discussed details of the drilling, drill-sampling, and quality control methods and procedures with the Integra technical team. Mr. Dyer has not visited the project site.

Section 13, Mineral Processing and Metallurgical Testing, was prepared under the supervision of Mr. Jack S. McPartland, Senior Metallurgist with ("McClelland") Laboratories, Inc., in Sparks, Nevada. Mr. McPartland visited the DeLamar project site on January 17, 2019.

Section 17 Recovery Methods was prepared under the supervision of Mr. Jeff Woods of Woods Process Services in Denver, Colorado, and Mr. John Welsh of Welsh Hagen Associates in Reno, Nevada. Mr. Welsh also contributed portions of Section 18 Infrastructure and Section 21 Capital and Operating Costs. Mr. Woods and Mr. Welsh are Qualified Persons under NI 43-101. Mr. Woods has not visited the project site. Mr. Welsh last visited the property on June 26, 2019.

The authors have reviewed the available data and have made judgments as to the general reliability of this information. Where deemed either inadequate or unreliable, the data were either eliminated from use or procedures were modified to account for lack of confidence in that specific information. Mr. Gustin and Mr. Weiss have made such independent investigations as deemed necessary in their professional judgment to be able to reasonably present the conclusions discussed herein.

The effective date of the current mineral resources that support the PEA is May 1, 2019, and the effective date of this technical report is September 9, 2019.

2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

In this report, measurements are generally reported in metric units. Where information was originally reported in Imperial units, conversions have been made with the following conversion factors:

Linear Measure

1 centimeter	= 0.3937 inch	
1 meter	= 3.2808 feet	= 1.0936 yard
1 kilometer	= 0.6214 mile	

Area Measure

1 hectare	= 2.471 acres	= 0.0039 square mile

Capacity Measure (liquid)

1 liter	= 0.2642 US gallons

Weight

1 tonne	= 1.1023 short tons	= 2,205 pounds
1 kilogram	= 2.205 pounds	

Conversion of Imperial to Metric Grades

1 troy ounce per short ton	= 34.2857 grams per metric tonne

Currency: Unless otherwise indicated, all references to dollars ($) in this report refer to currency of the United States.

Frequently used acronyms and abbreviations

AA	atomic absorption spectrometry
Ag	silver
Au	gold
cm	centimeters
core	diamond core-drilling method
^{o}C	degrees centigrade
CAD$	Canadian dollars
°F	degrees Fahrenheit
ft	foot or feet
g/t	grams per tonne
ha	hectares
ICP	inductively coupled plasma analytical method
in.	inch or inches
kg	kilograms
km	kilometers
ktpd	metric kilotonnes per day
kWh	kilowatt hour
l or L	liter
lbs	pounds
µm	micron
m	meters
Ma	million years old
mi	mile or miles
mm	millimeters

MW	megawatt
NSR	net smelter return
oz	ounce
ppm	parts per million
ppb	parts per billion
QA/QC	quality assurance and quality control
RC	reverse-circulation drilling method
RQD	rock-quality designation
t	metric tonne or tonnes
TPD	metric tonnes per day
tph	metric tonnes per hour
ton	Imperial short ton
U.S.	United States of America
XRD	x-ray diffraction
XRF	x-ray fluorescence

3.0 RELIANCE ON OTHER EXPERTS

Mr. Gustin, Mr. Weiss, Mr. Dyer and Mr. McPartland are not experts in legal matters, such as the assessment of the validity of mining claims, mineral rights, and property agreements in the United States or elsewhere. Furthermore, the authors did not conduct any investigations of the environmental, social, or political issues associated with the DeLamar project, and are not experts with respect to these matters. The authors have therefore relied fully upon information and opinions provided by Integra and Mr. Edward Devenyns, Mineral Land Consultant for Integra, with regards to the following:

- Section 4.2, which pertains to land tenure, including a Limited Due Diligence Review of the property prepared by Perkins Coie LLP (dated August 21, 2017) and further information from Perkins Coie LLP dated March 2, 2018 and March 8, 2018; and
- Section 4.3, which pertains to legal agreements and encumbrances.

The authors have relied fully upon information and opinions provided by Integra's consultant, Mr. Richard DeLong of EM Strategies, Inc., an expert in environmental and permitting matters. Section 4.4, which pertains to environmental permits and liabilities, was provided by Mr. DeLong in communications via emails on September 25, 2017 (DeLong, 2017) and July 17, 2019 (DeLong, 2019). Section 20 on environmental permitting, prepared by Mr. DeLong, was provided by Integra in a project communication via email dated September 11, 2019.

The authors have fully relied on Integra to provide complete information concerning the pertinent legal status of Integra and its affiliates, as provided in Sections 1, 2, and 4, as well as current legal title, material terms of all agreements, and material environmental and permitting information that pertains to the DeLamar project, as summarized in Sections 1 and 4.

4.0 PROPERTY DESCRIPTION AND LOCATION

The authors are not experts in land, legal, environmental, and permitting matters and express no opinion regarding these topics as they pertain to the DeLamar project. Subsections 4.2 and 4.3 were prepared under the supervision of Mr. Edward Devenyns, Mineral Land Consultant for Integra. Mr. Devenyns prepared a Limited Title Report on the unpatented claims dated August 15, 2017. A Limited Due Diligence Review of the property was prepared by Perkins Coie LLP dated August 21, 2017. On March 2 and March 8, 2018, Perkins Coie LLP provided MDA information concerning the Banner and Empire claims at Florida Mountain. Mr. Richard DeLong of EM Strategies, Inc., an expert in environmental and permitting matters, prepared Section 4.4.

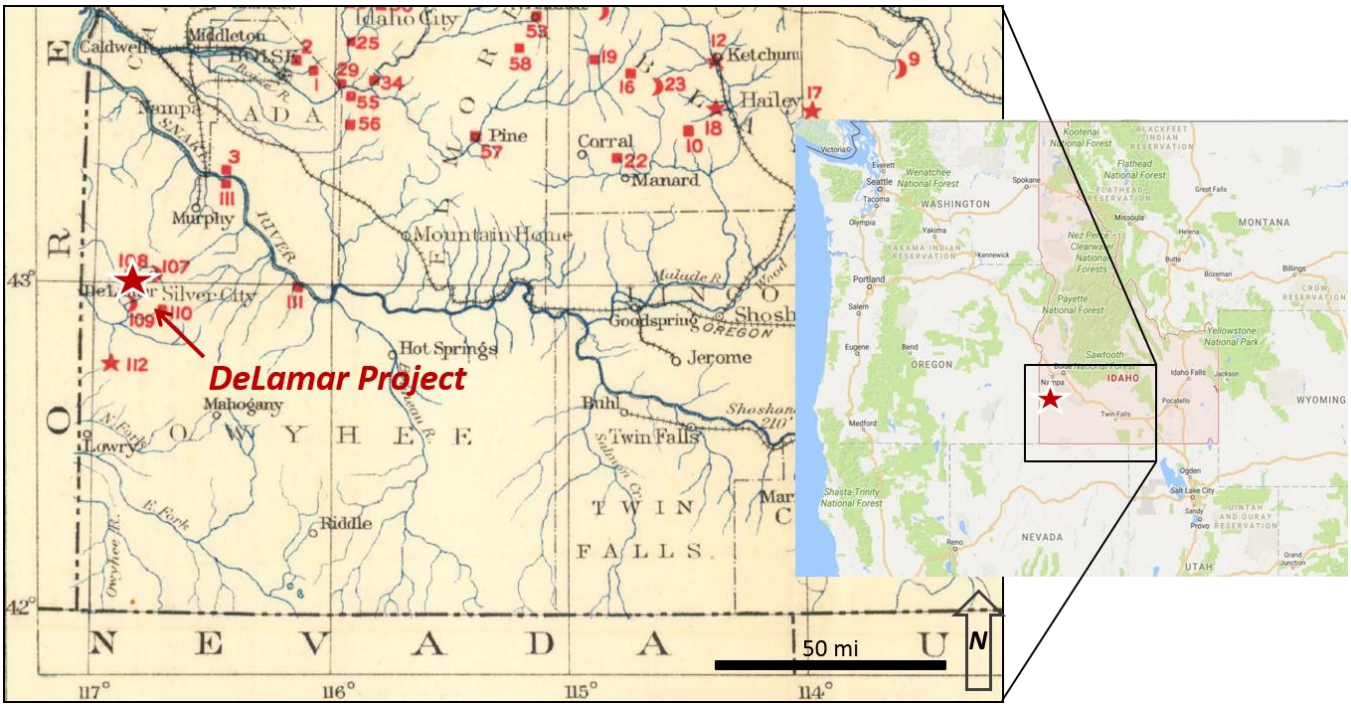
Integra owns 100% of the DeLamar project. All mineral titles are held by the DeLamar Mining Company ("DMC"), a wholly owned subsidiary of Integra.

Mr. Gustin and Mr. Weiss do not know of any significant factors or risks that may affect access, title, or the right or ability to perform work on the property, beyond what is described in this report.

4.1 Location

Integra's DeLamar gold-silver project is located in southwestern Idaho in Owyhee County, 80 kilometers southwest of the city of Boise, just west of the historical mining town of Silver City (Figure 4.1). The property is centered at approximately 43°00′48″N, 116°47′35″W, within the historical Carson mining district, and includes the formerly producing DeLamar silver-gold mine, which was last operated by the Kinross DeLamar Mining Company, a subsidiary of Kinross.

Figure 4.1 Location Map, DeLamar Gold – Silver Project
(modified from Hill and Lindgren, 1912; red numbers refer to 1912 mining districts)



4.2 Land Area

The DeLamar project consists of 748 unpatented lode, placer, and millsite claims, and 16 tax parcels comprised of patented mining claims, as well as certain leasehold and easement interests located in Owyhee County, Idaho. In total, the property covers approximately 8,100 hectares owned or controlled by Integra (Figure 4.2) and occupies portions of:

- Sections 30 and 31 of Township 4 South, Range 3 West;
- Sections 28, 29, and 31 through 36 of Township 4 South, Range 4 West;
- Sections 35 and 36 of Township 4 South Range 5 West;
- Section 6 and 7 of Township 5 South, Range 3 West;
- Sections 1 through 16 of Township 5 South, Range 4 West; and
- Sections 1 through 3, 10, 11, 14, 15 and 22 of Township 5 South, Range 5 West, Boise Base and Meridian.

A listing of the patented and unpatented claims and leasehold interests that comprise the property is provided in Appendix A, Parts 1 through 5. Integra represents that the list of claims and leasehold interests in Appendix A is complete to the best of its knowledge as of the effective date of this report. Included in Appendix A, Part 1, are five Idaho Department of Lands leases that are in the process of being issued to DMC.

DMC also owns mining claims and leases of State of Idaho lands located beyond the limits of the property described above. These landholdings are not part of the DeLamar project, although some of the claims are contiguous with those of the DeLamar and Florida Mountain claims and state leases.

Ownership of the unpatented mining claims is in the name of the holder (locator), subject to the paramount title of the United States of America, under the administration of the U.S. Bureau of Land Management ("BLM"). Under the Mining Law of 1872, which governs the location of unpatented mining claims on federal lands, the locator has the right to explore, develop, and mine minerals on unpatented mining claims without payments of production royalties to the U.S. government, subject to the surface management regulation of the BLM. Currently, annual claim-maintenance fees are the only federal payments related to unpatented mining claims, and these fees have been paid in full to September 1, 2020. The current annual holding costs for the DeLamar project unpatented mining claims are estimated at $132,245 (Table 4.1), including the county recording fees.

Other annual land holding costs, including county taxes for the patented claims and leased fee lands, and lease payments due to third-party claim owners, are listed in Table 4.1. The total annual land-holding costs are estimated to be $321,626.

Figure 4.2 Property Map for the DeLamar Project

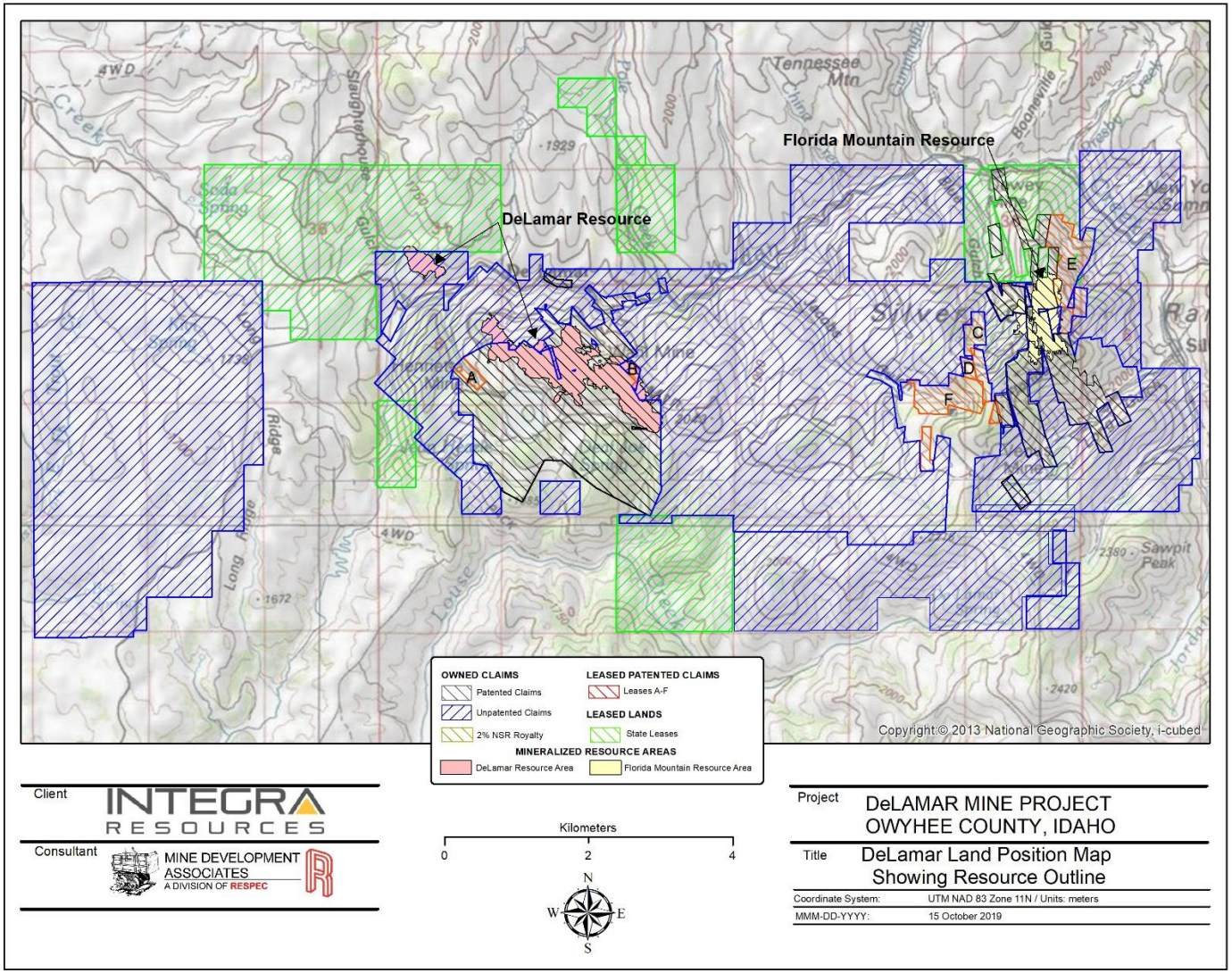


Table 4.1 Summary of Estimated Land Holding Costs for the DeLamar Project

Annual Fee Type	Amount
Unpatented Claims BLM Maintenance Fees	$ 132,165
Unpatented Claims County Filing Fees	$ 80
Estimated Holding Costs for Unpatented Mining Claims	**$ 132,245**
Access, Pipeline, Land Agreement Fees	$ 130,180
Owyhee County Patented Claims Taxes	$ 5,849
Patented Claims Agreement Fees	$ 30,100
State Lands Lease (annual rental and advanced minimum royalty payments)	$ 23,252
Total Estimated Annual Holding Taxes and Fees	**$ 321,626**

The reviews by Mr. Devenyns and Perkins Coie LLP have not identified any known fatal defects in the title of the claims, and the authors are not aware of any significant land use or conflicting rights, or such other factors and risks that might substantially affect title or the right to explore and mine the property, based on the information provided by Integra and Perkins Coie LLP.

Integra's subsidiary, the DeLamar Mining Company, holds the surface rights to the patented claims it owns and has leased, subject to various easements and other reservations and encumbrances. The DeLamar Mining Company has rights to use the surface of the unpatented mining claims for mining related purposes to September 1, 2020, and which it may maintain on a yearly basis beyond that by timely annual payment of claim maintenance fees and other filing requirements, and subject to the paramount title of the U.S. federal government. The DeLamar Mining Company holds surface rights to the areas it has under lease in accordance with the terms of each lease.

4.3 Agreements and Encumbrances

On November 3, 2017, Integra announced that it acquired 100% of the DeLamar gold – silver project from a wholly owned subsidiary of Kinross for CAD$7.5 million in cash and the issuance of Integra shares. In addition, Table 4.2 summarizes further the agreements and encumbrances applicable to the property. Fees other than royalties associated with these agreements are included in the land-holding costs of Table 4.1.

In terms of royalties, 101 of the 284 unpatented claims acquired from Kinross are subject to a 2.0% net smelter returns royalty ("NSR") payable to a predecessor owner (Table 4.2); this royalty is not applicable to the current project resources. There are also six lease agreements that include 2.5% to 5.0% NSR obligations (referred to as Leases A through F in Figure 4.2, and Party A through F in Table 4.2) that apply to 26 of the patented claims and one unpatented claim. These claims are located within portions of Sections 1, 2, 4, 6, 11, and 12 of Township 5 South, Range 4 West; Section 6 of Township 5 South, Range 3 West; Section 36 of Township 4 South, Range 4 West, and Section 31 of Township 4 South, Range 3 West, Boise Base and Meridian. Leases B and E apply to small portions of the DeLamar area (5% NSR to a maximum of $50,000) and Florida Mountain area (2.5% NSR to a maximum of $650,000) resources, respectively.

The property also includes 1,355 hectares (3,348 acres) leased from the State of Idaho under six separate State Mineral Leases that are subject to a 5.0% production royalty of gross receipts (Table 4.2), plus annual lease fees of $23,252 (Table 4.1). One of these leases has been issued and five are pending issuance. The issued lease has an expiration date of February 28, 2028. The five pending leases are expected to have expiration dates in the second half of 2029 (ten years from the date of issuance), depending on the actual date of issuance. The State of Idaho leases include very small portions of both the DeLamar and Florida Mountain resources.

Kinross has retained a 2.5% NSR royalty that applies to those portions of the DeLamar area claims acquired from Kinross that are unencumbered by the royalties described above. The Kinross royalty applies to more than 90% of the DeLamar area resources; the royalty will be reduced to 1.0% upon Kinross receiving total royalty payments of CAD$10,000,000.

A total of approximately 20% of the current Florida Mountain resources are subject to one or more of the royalties described above.

Figure 4.2 shows the areas subject to the royalties and lease agreements summarized in Table 4.2.

Table 4.2 Summary of Agreements and Encumbrances
(from Integra, 2019)

Owner	Number of Claims or Lease	Royalty
Kinross Gold	183 unpatented claims and 13 tax parcels comprised of patented claims	2.5% NSR up to CAD$10M; then 1.0% NSR
Predecessor Owner	101 unpatented claims	2.0% NSR
State of Idaho	3,348 acres under six separate Mining Leases	5.0% production royalty of gross receipts
Party A	1 patented claim	5.0% NSR to $50,000; then 2.5% NSR to a maximum of $400,000
Party B	1 patented claim	5.0% NSR to a maximum of $50,000
Party C	2 patented claims	2.5% NSR
Party D	1 patented claim	2.5% NSR
Party E	9 patented claims and 1 unpatented claim	2.5% NSR to a maximum of $650K
Party F	12 patented claims	2% NSR to a maximum of $400K

Portions of the property are subject to a private land agreement, road access agreement, pipeline agreement, State of Idaho Easement Agreement and a BLM right-of-way agreement that include lands and certain rights within portions Sections 2, 3, 4, 7, 9, 10, 11, 14 and 18 of Township 5 South, Range 4 West, and Sections 11, 12, 13, 14, 23, 24, 25 and 26 of Township 5 South, Range 5 West.

4.4 Environmental Liabilities and Permitting

The 1977 – 1998 DeLamar mine consisted of the DeLamar mine proper, as well as the Florida Mountain mining area. The DeLamar mine facilities, specifically the historical Sommercamp and North DeLamar open pits, incorporate essentially all the historical underground mining features (adits and dumps) in the vicinity. In the Florida Mountain area, many historical underground mining features remain to the north of the Florida Mountain open pits and waste rock dump, and several of these historical underground mining features are located within the DeLamar project, including collapsed adits, dumps, and collapsed structures. None of these features have water draining from them.

The DeLamar mine has been in closure since 2003. Since 2003, the following reclamation and closure activities have been conducted on the DeLamar project:

- Tailings pond de-watered and capped with clay and soil;
- Two waste piles regraded and capped with clay and soil;
- Heap-leach pad removed;
- Much of the reclaimed surface includes an engineered cover consisting of two feet (61 centimeters) of compacted clay, 10 inches (25.4 centimeters) of non-acid generating run-of-mine ("ROM") material, and 8 inches (20.3 centimeters) of suitable plant growth media;
- The DeLamar mine facilities include three primary pit areas. These are the North DeLamar, Sommercamp – Regan (including North and South Wahl), and Glen Silver pits (Figure 6.2), which are partially backfilled and clay capped to allow for positive drainage;
- The Florida Mountain mine facilities within the DeLamar project include the Jacobs Gulch waste-rock dump, which has been regraded and reclaimed, and the Tip-top, Stone Cabin, and Black Jack pits, which have been partly back-filled;
- The DeLamar mine is in the Closure Phase with the Idaho Department of Lands ("IDL") and activities that focus on water management;
- Water management includes collection of water at four primary collection and pumping stations referred to as Meadows, SP5, Spillway, and SP1. There are also two ancillary pumping stations at Adit 16 and SP14; and
- The collection stations route water to a primary lime amendment facility and a smaller caustic-drip facility. Water passing through the lime amendment plant is routed to a storage pond and seasonally released at a nearby land application site ("LAS").

The DeLamar project holds the following primary permits: two Plans of Operation ("PoO"), one with IDL and the BLM (PoO #248), and one with IDL (PoO #936). In addition, the DeLamar Mining Company holds a Cyanidation Permit from the Idaho Department of Environmental Quality ("IDEQ"), an Air Quality Permit from IDEQ, a Dam Safety Permit from the Idaho Department of Water Resources ("IDWR"), and a 2015 Multi-Sector General Permit ("MSGP"), Storm Water Permit, and a Ground Water Remediation Permit from the United States Environmental Protection Agency ("EPA").

Even though a substantial amount of reclamation and closure work has been completed at the site, there remain ongoing water-management activities and monitoring and reporting. The monitoring and reporting activities include: stream water quality and benthic, air quality, the LAS, and quality assurance and control. Water-management activities consist of an annual cycle of winter and spring storage and then summer and fall treatment and land application discharge.

In January of 2017, Kinross submitted to IDL a reclamation bond reduction request, prepared by SRK Consulting (US) Inc. IDL responded in writing on April 24, 2017, indicating they had received the partial bond reduction request on March 29, 2017, and stated that they needed more time to complete the required site inspection prior to acting on the bond reduction request. On May 31, 2017, the IDWR issued a letter stating their relinquishment of any claims on the bond held by IDL. On June 19, 2017, IDL concurred with Kinross' request for a $9,032,148 reduction in the bond. A reclamation bond of $2,778,929 remains with the Idaho Department of Lands ("IDL") and a reclamation bond of $100,000 remains with the Idaho Department of Environmental Quality ("IDEQ"). In addition, a reclamation bond in the amount of $51,500 has been placed with the BLM to cover exploration activities on public lands.

As of the date of this report, Integra is conducting a reverse-circulation ("RC") and core drilling program on patented and unpatented mining claims in the DeLamar and Florida Mountain areas of the project. This drilling is being undertaken under a Notification from IDL, as well as two Notices filed with the BLM. The exploration program recommended in Section 25.0 includes proposed drilling in the Florida Mountain area of the project, as well as further drilling in the DeLamar area. This proposed work would necessitate a modification to the existing Notification for drilling in the DeLamar area, and a new Notification for Florida Mountain drilling performed on patented claims. A Notice would need to be filed with the BLM if any of the recommended drilling is undertaken on unpatented claims. Separate Notices would be filed with the BLM for each of the DeLamar and Florida Mountain areas of unpatented claims.

The authors are not aware of any significant factors and risks that may affect access, title, or the right or ability to perform work on the property, other than those discussed above.

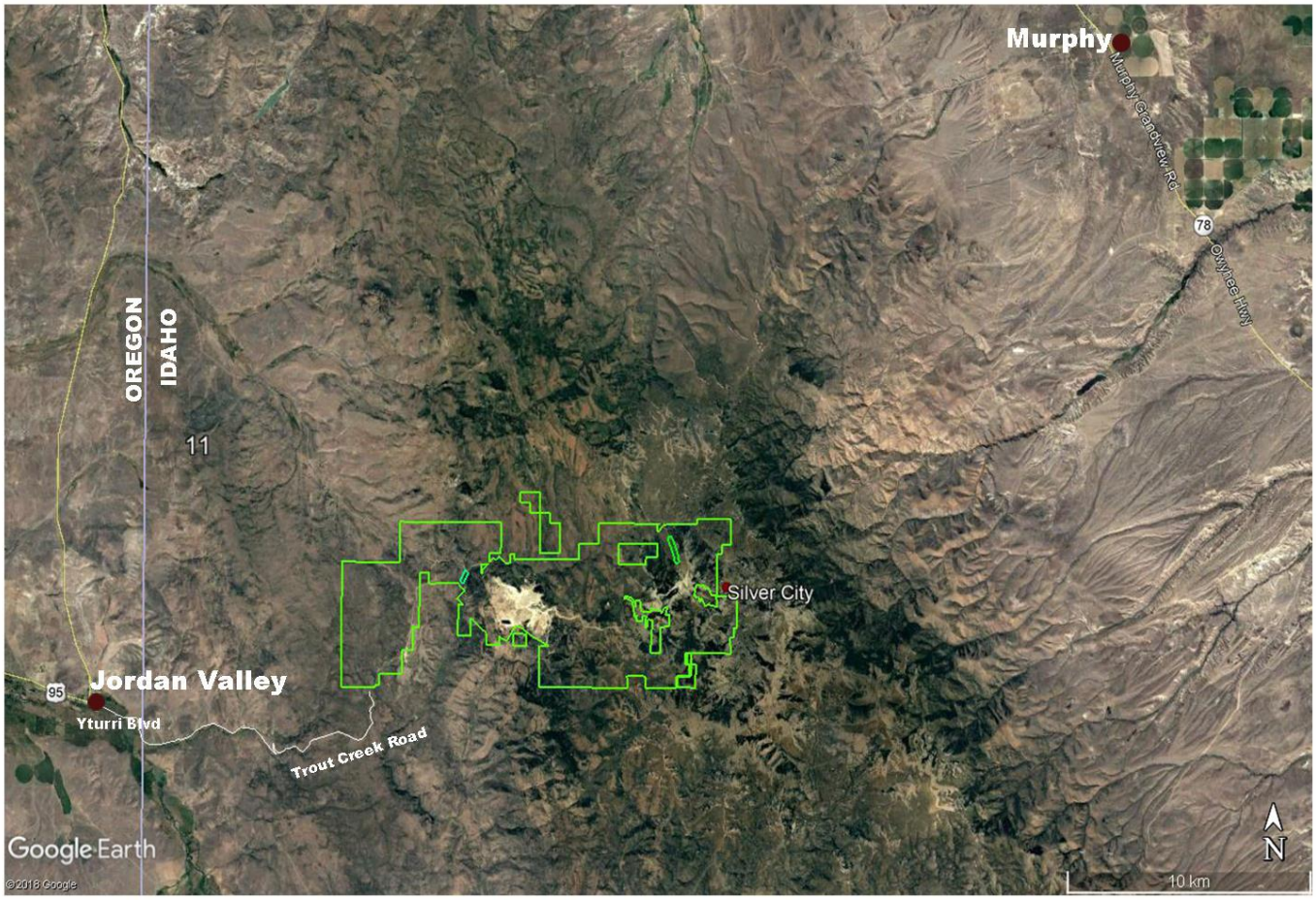
5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The information summarized in this section is derived from publicly available sources, as cited. Mr. Gustin and Mr. Weiss have reviewed this information and believe this summary is materially accurate.

5.1 Access to Property

The principal access is from U.S. Highway 95 and the town of Jordan Valley, Oregon, proceeding east on Yturri Blvd. from Jordan Valley for 7.6 kilometers to the Trout Creek Road (Figure 5.1). It is then another 39.4 kilometers travelling east on the gravel Trout Creek Road to reach the DeLamar mine tailings facility and nearby site office building. Travel time by automobile via this route is approximately 35 minutes. Secondary access is from the town of Murphy, Idaho and State Highway 78 (Figure 4.1 and Figure 5.1), via the Old Stage Road and the Silver City Road. Travel time by this secondary route is estimated to be about 1.5 hours.

Figure 5.1 Access Map for the DeLamar Project
(produced by MDA, 2019)



5.2 Physiography

The property is situated in rolling to mountainous terrain of the Owyhee Mountains at elevations ranging from about 1,525 meters to 2,350 meters above sea level within portions of the De Lamar, Silver City, Flint, and Cinnabar Mountain U.S.G.S. 7.5-minute topographic quadrangles. Portions of the property are forested with second- or third-growth spruce, pine, aspen, and fir. Vegetation types include Douglas fir, juniper – mountain mahogany, sagebrush, mixed shrubs, and wyethia meadow communities.

5.3 Climate

The climate can be described as moderately arid in the lower elevations to mid-continental at the higher elevations, with warm summers and cold, snowy winters. MDA is unaware of published historical temperature and precipitation data for the Owyhee Mountains. According to Kinross' DeLamar mine personnel, summer maximum temperatures can reach 20°C and winter minimum temperatures can be as low as -40°C. Precipitation at the mine site is believed to average about 50 centimeters per year, most of which occurs as winter snowfall. Snow cover at the upper elevations can be 1.0 to 2.0 meters deep. Mining operations have been demonstrated to be feasible year-round but do require snow removal equipment to maintain road access during the winter. Road access for exploration may be limited or interrupted by snow during December through April.

5.4 Local Resources and Infrastructure

A highly trained mining and industrial workforce is available in Boise, Idaho, approximately 100 kilometers northeast of the project area. The project area is served by U.S. Interstate Highway 84 through Boise and by U.S. Highway 95 about 30 kilometers west of the site in southeastern Oregon. Mining and industrial equipment, fuel, maintenance, and engineering services and supplies are available in Boise, Idaho, as are telecommunications, a regional commercial airport, hospitals, and banking.

Housing, fuel, and schools are available in the nearby town of Jordan Valley, Oregon, which presently has a population of about 175 inhabitants. There are as many as a few dozen summer residents of the old historical mining town of Silver City, located about 8.5 kilometers east of the DeLamar mine, but few or no residents during the winter when road access is interrupted by accumulated snow.

An administrative office building with communications and an emergency medical clinic from the historical, late 20th century open-pit mining operation remain on site and in use. A truck shop and storage building also remain on site. The processing plant and facilities, crushing equipment, and assay laboratory have been removed from the property. Electrical power at the project site is delivered via a 69Kv transmission line from the Idaho Power Company. Although the project area is generally hilly, flat areas are present and have served in the past for siting the processing plant and tailings storage areas. Developed water wells are present for mining and process requirements.

6.0 HISTORY

The information summarized in this section has been extracted and modified to a significant extent from Piper and Laney (1926), Asher (1968), Bonnichsen (1983), Thomason (1983), and unpublished company files, as well as other sources as cited. Mr. Gustin and Mr. Weiss have reviewed this information and believe this summary is materially accurate.

For clarity, this report will retain the term "De Lamar" to refer to the historical De Lamar underground mining operation of the late 19th and early 20th centuries and, consistent with official USGS topographic maps and place names, the historical De Lamar town site on Jordan Creek and De Lamar Mountain. According to Bonnichsen (1983), the present-day term "DeLamar" follows the usage of Earth Resources Company starting in the 1970s (see below). In this report, the term "DeLamar mine" refers to the open-pit mine and processing operation at De Lamar Mountain that began in the late 1970s.

6.1 Carson Mining District Discovery and Early Mining: 1863 – 1942

Mining activity began in the DeLamar project area in May of 1863 when placer gold deposits were discovered in Jordan Creek, just upstream from what later became the town site of De Lamar (Wells, 1963 as cited in Asher, 1968). The placer deposits were traced up stream, beyond the DeLamar project area, and during the summer of 1863 the first silver-gold lodes were discovered in quartz veins at War Eagle Mountain, which is outside the current property controlled by Integra. This resulted in a rush of miners to the area and the initial settlement of Silver City. Several small mines at War Eagle Mountain were quickly developed with rich, near-surface ore. By 1866, there were 12 mills in operation (Piper and Laney, 1926). Grades decreased at depth and in 1875 the Bank of California failed, resulting in a loss of financial backing, which contributed to the closure of the mines by 1876. According to Lindgren (1900), cited in Bonnichsen (1983) and Piper and Laney (1926), an estimated $12 to $12.5 million was produced from the War Eagle Mountain veins from 1863 through 1875, or the equivalent of 600,000 to 625,000 ounces of gold. Silver-to-gold ratios of the ores during this period were on the order of 1:1 to 1:6 according to Piper and Laney (1926).

The general area of De Lamar, Florida Mountain, Silver City and War Eagle Mountain was known as the Carson mining district, which was larger than the current property controlled by Integra. There was only minor production from sporadic activity in the district at the War Eagle Mountain mines from 1876 through 1888, and some of the mines were never reopened. However, significant silver-gold veins were discovered during this time period at De Lamar Mountain and at Florida Mountain. Captain J.R. De Lamar founded the De Lamar Mining Company and was largely responsible for the development of important veins at the original, underground De Lamar mine, just to the south of Jordan Creek. De Lamar's name was applied to the mine, the mountain, and the small mining town that was established on Jordan Creek.

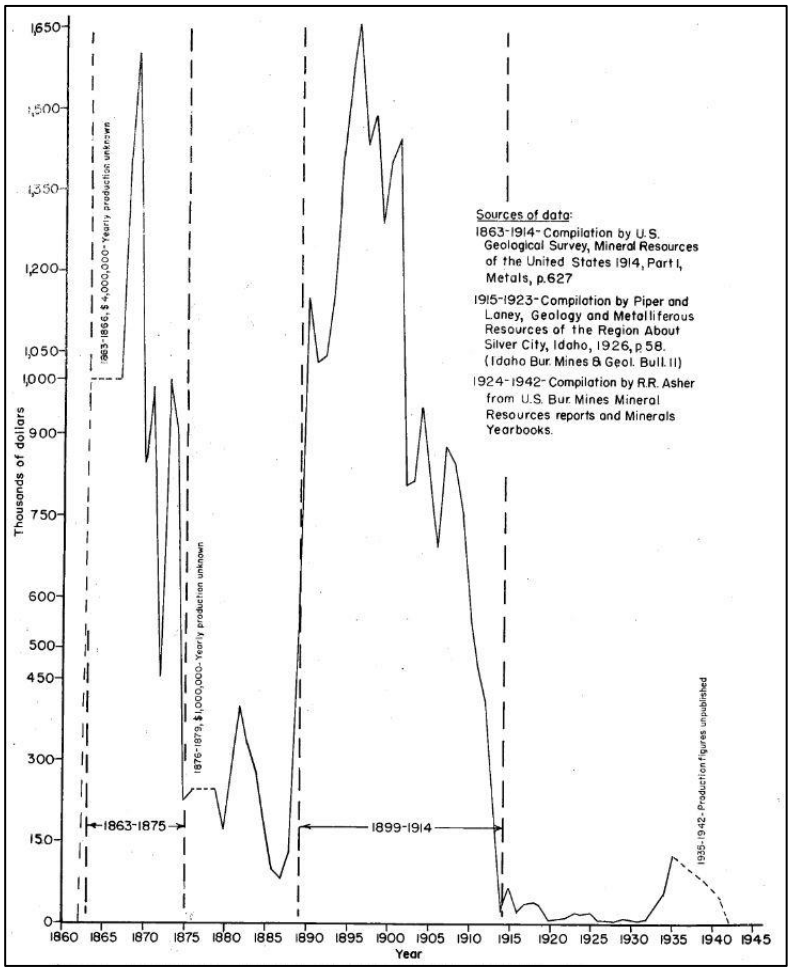
In 1889, rich ore shoots were discovered in veins at the De Lamar mine area. De Lamar sold his interest to the London-based DeLamar Mining Company, Ltd. in 1901. Declining grades and increasing costs caused the closure of the De Lamar mines by 1914. An estimated total production value of precious metals of nearly $23 million was reported from the Carson district for the period 1889 – 1914 by Piper and Laney (1926). The De Lamar mine is believed to have produced approximately 400,000 ounces of gold and 5.9 million ounces of silver from a minimum of about 726,000 tonnes milled from 1891

through 1913, based on annual company reports (Gierzycki, 2004a). Mines in Florida Mountain are estimated to have produced a total of 133,000 ounces of gold and 15.4 million ounces of silver from 1883 to 1910 (Bonnichsen et al. undated, cited in Gierzycki, 2004a).

Very little production took place in the Carson district until the 1930s, when gold and silver prices increased. Placer gold was recovered from Jordan Creek from 1934 to 1940, and in 1938 a 181 tonne-per-day flotation mill was constructed to process dumps from the De Lamar mine. The flotation mill reportedly operated until the end of 1942. In 1939, the Morrison-Knudson Company excavated a small open pit on the east side of Florida Mountain, but the operation was not profitable and was shut down in November of that year (Asher, 1968).

A summary of estimated annual production value for the entire district, including the DeLamar project, through 1942 is shown in Figure 6.1. Altogether, the district is believed to have produced about 1 million ounces of gold and 25 million ounces of silver from 1863 through 1942 (Piper and Laney, 1926; Bergendahl, 1964). Gierzycki (2004b) estimated a total district production of 0.6 million ounces of gold and 42 million ounces of silver for this period.

Figure 6.1 Estimated Annual Production Value, Silver City (Carson) Mining District 1863-1942
(from Asher, 1968)



6.2 Historical Exploration Since the 1960s

It is believed that mining properties in the De Lamar project area were largely inactive from 1942 until the mid-1960s. Anecdotal information suggests that the Sidney Mining Company and the Continental Materials Corporation ("Continental") both engaged in diamond-core ("core") drilling in 1966, but MDA has information only for the Continental drilling during this time. Continental's holes were drilled to test veins down-dip from stopes of the old De Lamar mine (Porterfield, 1992).

During the late 1960s, the district began to undergo exploration for near-surface, bulk-mineable gold-silver deposits, but few records of the work are available. The Glen Silver Mining Company conducted core drilling in what later became either the Glen Silver or the Sommercamp area of the DeLamar project, but the exact locations of the drill holes are not known to MDA.

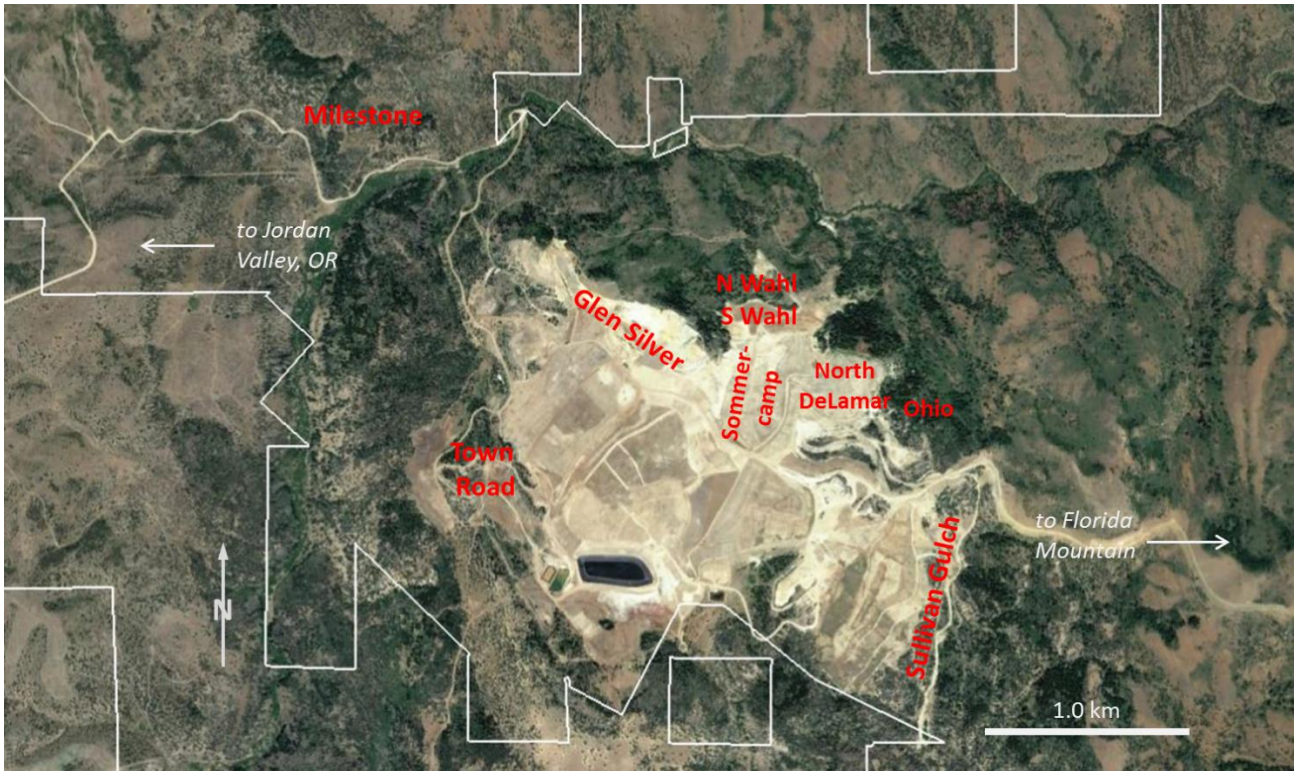
In 1969, the "Silver Group" was formed as a joint venture comprised of Earth Resources Company ("Earth Resources"), Superior Oil Company, and Canadian Superior Mining (U.S.) Ltd. The Silver Group acquired property in the De Lamar – Florida Mountain area and conducted geological mapping and sampling. Much of the early exploration work was carried out by Perry, Knox, Kaufman Inc. for Earth Resources, the operator of the project.

During 1969 and 1970, Earth Resources carried out trenching, sampling, and surface geological work, and drilled 39 conventional rotary drill holes at De Lamar Mountain. This resulted in the discovery of broad areas of near-surface silver-gold mineralization in the Sommercamp and Glen Silver zones, and what Earth Resources termed the North DeLamar zone. Following these discoveries, Earth Resources ramped up exploration and development drilling, and from about 1971 through 1976 at least 432 holes were drilled, mainly in the North DeLamar, Glen Silver, Sommercamp – Regan (including North and South Wahl), and Ohio areas (Figure 6.2). This drilling also included the first holes drilled at the nearby Sullivan Gulch and Milestone prospects, as well in the Florida Mountain area.

The Sidney Mining Company drilled eight core holes in the Sommercamp and North DeLamar zones in 1972. In 1974, Perry, Knox, Kaufman Inc. completed a feasibility study for the Silver Group with reserve estimates for an open-pit mining scenario at the Sommercamp and North DeLamar zones. In 1977, Earth Resources commenced operation of the DeLamar silver-gold mine with initial open-pit mining at the North DeLamar and Sommercamp zones (see Section 6.3 for a summary of the DeLamar mine production). In 1981, Earth Resources was acquired by the Mid Atlantic Petroleum Company ("MAPCO"), and Earth Resources continued to operate the DeLamar mine and exploration joint venture.

Earth Resources continued to explore the Sullivan Gulch, North DeLamar, Glen Silver and Florida Mountain zones between 1978 and mid-1984. Incomplete records show that at least 135 holes were drilled by Earth Resources in these areas of the property.

Figure 6.2 Aerial View, Zones of Exploration and Mining Since 1969 within the DeLamar Area
(produced by MDA, 2019)



Note: North and South Wahl are included in what is referred to as the Sommercamp – Regan zone.

In September of 1984, the NERCO Minerals Company Inc. ("NERCO") purchased MAPCO's interest in the DeLamar project and became the operator of the joint venture. Less than a year later, in mid-1985, NERCO purchased the interests of the remaining joint venture partners and thereby attained 100% ownership of the project.

During 1985 through 1992, NERCO conducted extensive exploration and development drilling, as well as surface mapping and sampling. Drilling was focused mainly on expansion and definition of bulk-mineable mineralization at Florida Mountain, with significant amounts of drilling also completed at North DeLamar, Glen Silver, Sullivan Gulch, Town Road, and Milestone. Incomplete records indicate that a minimum of 1,594 holes were drilled by NERCO within the DeLamar project during this period.

NERCO was purchased by the Kennecott Copper Corporation ("Kennecott"), then a subsidiary of Rio Tinto – Zinc Corporation ("RTZ"), in 1993. Two months later in 1993, Kennecott sold its 100% interest in the DeLamar mine and property to Kinross.

Kinross continued exploration of the property while operating the DeLamar mine. A total of 338 exploration and development holes were drilled by Kinross in 1993 through 1997. Most of the drilling was focused in the Glen Silver, North DeLamar, and Florida Mountain areas of the project.

In addition to the surface sampling, drilling, and geological work, several campaigns of geophysical studies were performed at various times in the project history.

Kinross ceased exploration work in 1997 and mining was halted at the end of 1998 due to unfavorable metal prices. In 1999, milling ceased and Kinross placed the DeLamar and Florida Mountain operations on care and maintenance. Mine closure activities commenced in 2003. Mine closure and reclamation were nearly completed by 2014, including removal of the mill and other mine buildings, and drainage and cover of the tailings facility.

The property continued to be in closure and monitoring from 2014 to 2017.

6.3 Modern Historical Mining: 1977 through 1998

Total open-pit production from 1977 through 1998, including the Florida Mountain operation, is estimated at approximately 750,000 ounces of gold and 47.6 million ounces of silver (Gierzycki, 2004b). Although the mill reportedly continued to operate for some unknown amount of time in 1999, historical production records are only available to the end of 1998.

Earth Resources commenced open-pit operations and milling at the DeLamar mine in 1977. The mine initially operated five days per week with a target production of about 9,980 tonnes per day of ore and waste. Ore was processed by grinding in ball mills followed by agitated tank leaching with cyanide prior to precipitation with zinc dust. By the late 1980s, NERCO was mining ore and waste that totaled 21,772 tonnes per day and the mill processing capacity was 1,996 tonnes per day. At the time of the Kinross acquisition in 1993, the DeLamar mine was operating at a mining rate of 27,216 tonnes per day and a milling capacity of about 3,629 tonnes per day (Elkin, 1993). The DeLamar mine produced 421,300 ounces of gold and about 26 million ounces of silver from about 12.9 million tons mined from start-up in 1977 through to the end of 1992 (Table 6.1). Production during this period came from pits developed in the Glen Silver, Sommercamp – Regan, and North DeLamar areas.

Kinross commenced production at Florida Mountain in 1994, while continuing operations at the DeLamar mine, moving Florida Mountain ore to the DeLamar mill via an 8.4-kilometer haul road. Material was excavated from three open pits on the west side of the crest of Florida Mountain from 1994 through 1998. These were named the Stone Cabin, Tip Top, and Black Jack pits (Figure 6.3 and Figure 6.4). The Florida Mountain operation was formally referred to as the Stone Cabin mine in permitting and other documents. Gierzycki (2004b) estimated that 124,500 ounces of gold and 2.6 million ounces of silver were produced from the Stone Cabin mine in 1994 through the end of mining in 1998, based on an examination of files and company reports at the DeLamar mine

Mining in the Glen Silver – Sommercamp – North DeLamar areas continued simultaneously with the Florida Mountain operation. It has been reported that 625,500 ounces of gold and 45 million ounces of silver were produced from the Glen Silver – Sommercamp – North DeLamar areas over the entire life of mine from 1977 through 1998 (Gierzycki, 2004b).

Table 6.1 DeLamar Mine Gold and Silver Production 1977 – 1992

(from Elkin, 1993)

Year	Ore	Mill Grade		Bullion Poured	
	(short dry tons)	Gold	Silver	total troy ounces	
		(oz/ton)	(oz/ton)	Gold	Silver
1977	309,000	0.034	3.55	9,600	853,000
1978	637,000	0.031	3.78	18,100	1,872,000
1979	715,000	0.034	3.12	22,200	1,734,000
1980	780,000	0.031	2.53	22,100	1,534,000
1981	771,000	0.034	2.55	24,000	1,529,000
1982	738,000	0.036	2.77	24,300	1,589,000
1983	846,000	0.035	2.32	27,100	1,526,000
1984	784,000	0.023	2.83	15,500	1,742,000
1985	820,000	0.038	2.66	29,800	1,751,000
1986	849,000	0.035	2.52	27,700	1,713,000
1987	861,000	0.037	2.54	30,200	1,738,000
1988	830,000	0.033	2.34	32,000	1,738,000
1989	840,000	0.033	2.56	34,000	1,863,000
1990	829,000	0.037	2.04	30,400	1,374,000
1991	1,117,000	0.035	1.99	36,700	1,702,000
1992	1,156,000	0.035	2.01	37,600	1,820,000

Figure 6.3 Aerial View of the Florida Mountain (Stone Cabin Mine) Area

(produced by MDA, 2019)

Figure 6.4 Photograph of the Reclaimed Florida Mountain (Stone Cabin) Mine Area

(view looking south-southeast)

6.4 Historical Resource and Reserve Estimations

The estimates described in this subsection are presented herein as an item of historical interest with respect to historical open-pit mining and exploration at the DeLamar property. The historical estimations presented below are considered relevant because they represent an "ore reserve" that formed the basis of the initial open-pit mining, "reserves" estimated at the time of Kinross' acquisition of the mining operations, and "resources" estimated at the time of closure of the open-pit mining operations. The classification terminology is presented as described in the original references, but these categories do not conform to the measured, indicated, and inferred mineral resource classifications as set out in NI 43-101 and the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Definition Standards). There is insufficient information for Mr. Gustin to understand how these historical categories differ from CIM Definition Standards. In addition, Mr. Gustin has not completed sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and Integra is not treating these historical estimates as current mineral resources or mineral reserves. These historical estimates have been superseded by the current mineral resources described in this report and therefore they cannot be upgraded or verified as current mineral resources or reserves. Accordingly, these estimates are relevant only for historical context and should not be relied upon. The current mineral resources for the DeLamar project are discussed in Section 14.0.

The first reported historical "ore reserve" was presented in a 1974 feasibility study prepared by the Exploration Division of Earth Resources. A total of 4.124 million tonnes of "ore reserves" with average grades of 142.29 grams Ag/t and 1.58 grams Au/t, for about 18.8 million silver ounces and 210,000 gold

ounces, were estimated for the Sommercamp and North DeLamar zones as shown in Table 6.2 (Earth Resources, 1974).

At the time of the Kinross acquisition of the DeLamar operations and properties in 1993, the end-of-year 1992 reserves for the DeLamar mine area were estimated by Elkin (1993) at approximately 9.335 million tonnes with average silver and gold grades of 55.86 grams Ag/t and 0.72 grams Au/t, respectively (Table 6.2). Following the cessation of mining at the end of 1998 due to low metal prices, Kinross reported estimated resources and no reserves of 8.406 million tonnes with average silver and gold grades of 32.05 grams Ag/t and 1.25 grams Au/t, respectively (Table 6.2). The historical resources presented in Table 6.2 are based on the drill data available at the time of the estimations; the drill data are discussed in Sections 10.0, 11.0, 12.0, and 14.2.1.

Table 6.2 Historical Resource and Reserve Estimates

Year	Company	Area	Classification	Tonnes (millions)	Ag Grade g/tonne	Au Grade g/tonne	Ag Oz (millions)	Au Oz (millions)	Cutoff Grade
1974	Earth Resources	Sommercamp	"ore reserves"	2.312	178.63	1.06	13.3	0.08	2.0 oz/ton Ag Eq
	Earth Resources	North Delamar	"ore reserves"	1.813	95.66	2.23	5.6	0.13	2.0 oz/ton Ag Eq
1974		***total***		**4.124**	**142.16**	**1.58**	***18.8***	***0.21***	
EOY 1992	Kinross[1,2]	Glen Silver	"P&P mill"	3.958	53.83	0.82	6.848	0.105	2.5 oz/ton Ag Eq
		Glen Silver	"P&P low grade"	2.186	30.17	0.51	2.121	0.036	1.8 oz/ton Ag Eq
		South Wahl	"P&P mill"	0.524	79.54	1.75	1.341	0.029	2.5 oz/ton Ag Eq
		South Wahl	"P&P low grade"	0.019	50.40	0.34	0.031		1.8 oz/ton Ag Eq
		Sommercamp/Regan	"P&P mill"	0.678	152.23	0.69	3.317	0.015	2.5 oz/ton Ag Eq
		Sommercamp/Regan	"P&P low grade"	0.318	42.17	0.38	0.432	0.004	1.8 oz/ton Ag Eq
		Stone Cabin	"P&P mill"	7.795	28.80	1.82	7.194	0.454	0.03 oz/ton Au Eq
		Stone Cabin	"P&P low grade"	4.050	15.43	0.65	2.008	0.086	0.02 oz/ton Au Eq
		Stockpile	"P&P mill"	0.422	70.97	0.65	0.963	0.009	
		Stockpile	"P&P low grade"	0.205	44.23	0.38	0.292	0.002	
		Ore Pad	"P&P mill"	0.244	67.89	0.89	0.533	0.007	
		Ore Pad	"P&P low grade"	0.780	34.63	0.48	0.869	0.012	
		total	*"P&P mill"*	13.620	46.29	1.41	*20.196*	*0.619*	
		total	*"P&P low grade"*	7.559	23.66	0.58	*5.753*	*0.14*	
	total 'P&P mill + low grade"			**21.179**	**38.06**	**1.13**	***25.949***	***0.759***	
EOY 1997	Kinross[3]	all	"P&P"	7.688	36.04	1.23	8.907	0.304	
		all	"Possible Reserves"	0.766	28.27	1.18	0.767	0.032	
		total all	**"P&P + Possible"**	**8.454**	**35.34**	**1.23**	**9.674**	**0.336**	
EOY 1998	Kinross[4]	***all***	***"Measured, Indicated and Inferred Resources"***	**8.406**	**32.05**	**1.25**	***9.547***	***0.372***	

notes: EOY = year ending on December 31; "P&P" = Proven and Probable Reserves

[1] Elkin (1993); in place, mineable, partially diluted, metalurgical recovery not applied

[2] Elkin (1993); price assumed = $360/oz of gold and $4.00/oz of silver for DeLamar;
price assumed = $380/oz of gold and $4.20/oz of silver for Stone Cabin

[3] Kinross Gold Corporation Annual Report for 1997

[4] Kinross Gold Corporation Annual Report for 1998

Mr. Gustin has not done sufficient work to classify the historical estimates summarized in Table 6.2 as current mineral resources or mineral reserves, which are relevant only for historical context, and Integra is not treating these historical estimates as current mineral resources or mineral reserves. Mr. Gustin is unaware of the key assumptions, parameters, and methods used to prepare the historical estimates. Accordingly, these estimates should not be relied upon. The current mineral resources for the DeLamar project are discussed in Section 14.0.

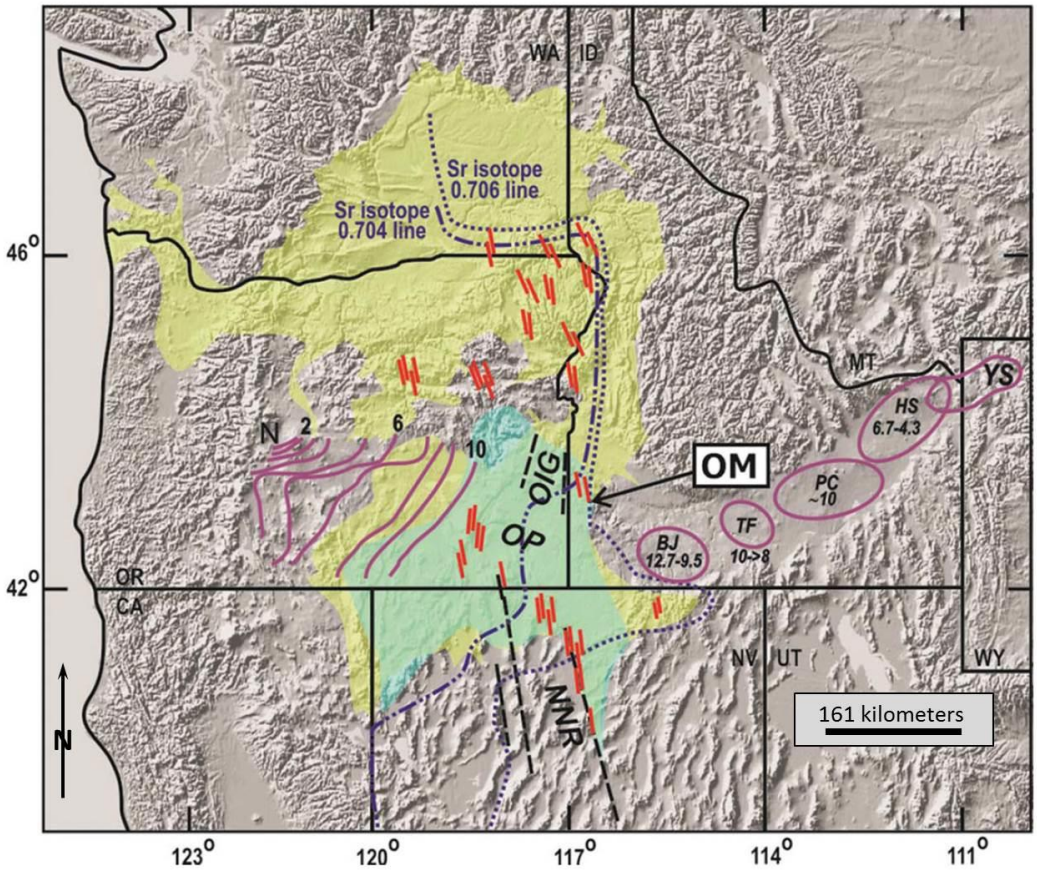
7.0 GEOLOGIC SETTING AND MINERALIZATION

The information presented in this section of the report is derived from multiple sources, as cited. Mr. Gustin and Mr. Weiss have reviewed this information and believe this summary accurately represents the DeLamar project geology and mineralization as it is presently understood.

7.1 Regional Geologic Setting

The DeLamar project is situated in the Owyhee Mountains, which are located near the east margin of the mid-Miocene Columbia River – Steens flood basalt province and the west margin of the Snake River Plain (Figure 7.1). The geology of various parts of the Owyhee Mountains has been described by Lindgren and Drake (1904), Piper and Laney (1926), Asher (1968), Bennett and Galbraith (1975), Panze (1975), Ekren et al. (1981), Ekren et al. (1982), and Bonnichsen and Godchaux (2006). As summarized by Bonnichsen (1983), Halsor et al. (1988), and Mason et al. (2015), the Owyhee Mountains comprise a major mid-Miocene eruptive center, generally composed of mid-Miocene basalt flows and younger, mid-Miocene rhyolite flows, domes and tuffs, developed on an eroded surface of Late Cretaceous granitic rocks. This Miocene magmatic and volcanic activity coincided with the regional Columbia River – Steens flood basalt event at about 16.7 to ~14.5 Ma (Mason et al., 2015).

Figure 7.1 Shade Relief Map with Regional Setting of the Owyhee Mountains
(from Mason et al., 2015)

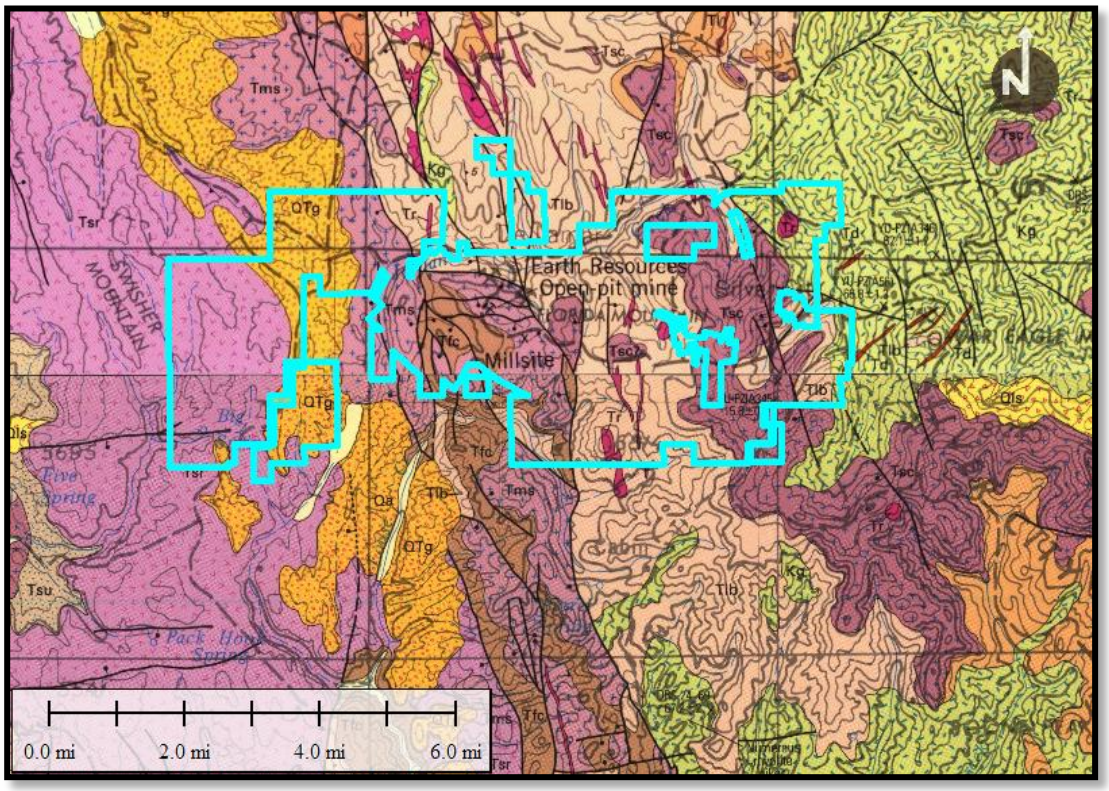


Note: OM = Owyhee Mountains; OP = Oregon Plateau; OIG = Oregon-Idaho graben; NNR = Northern Nevada Rift. Yellow shading shows the Columbia River – Steens flood basalt province; green shading indicates the Oregon Plateau underlain mainly by mid-Miocene silicic volcanic rocks. Red lines show eruptive loci and dike swarms; purple lines and ovoids are isochrons and silicic volcanic centers, respectively, with ages of silicic volcanism of the Oregon High Lava Plains and Snake River – Yellowstone provinces in Ma. Dark blue dashed and dotted lines are strontium isopleths. See Mason et al. (2015) for sources of data.

7.2 Owyhee Mountains and District Geology

Five informal rock-stratigraphic sequences have been defined in the central Owyhee Mountains and the De Lamar – Silver City area (Figure 7.2). From oldest to youngest these are the 1) Late Cretaceous Silver City granite; 2) mid-Miocene lower basalt; 3) mid-Miocene latite and quartz latite; 4) mid-Miocene Silver City rhyolite; and 5) mid-Miocene Swisher Mountain Tuff (formerly tuff of Swisher Mountain). The Silver City granite crops out near the crest and in the eastern part of the range (Figure 7.2), and it forms the pre-volcanic basement in the area. It has been described as mainly medium- to coarse-grained biotite-muscovite granodiorite to quartz monzonite and albite granite (e.g., Bonnichsen, 1983). It is considered to represent an outlying portion of the Idaho Batholith based on Late Cretaceous potassium-argon age dates, and similarities in composition, and mineralogy (Taubeneck, 1971; Panze, 1972).

Figure 7.2 Geologic Map of the Central Owyhee Mountains
(from Ekren et al., 1981)



The Silver City granite is directly overlain by flows of the Miocene lower basalt, which have filled up to several hundreds of feet of relief on the granite. This demonstrates that the Silver City granite had been

exhumed and underwent subaerial erosion by mid-Miocene time. The lower basalt is exposed in a northwest-trending band through the central part of the Owyhee Mountains (Figure 7.2) and consists of as much as 762 meters of flows of alkali-olivine to tholeiitic basalt that change upward to basaltic andesite and trachyandesite (Asher, 1968; Ekren et al., 1982; Bonnichsen, 1983; Thomason, 1983). As pointed out by Bonnichsen (1983), these basalts were erupted between 17 and 16 Ma, recalculated with modern decay constants from age dates of Panze (1975) and Armstrong (1975), and the lower part of the basalt sequence includes flows with distinctive large plagioclase phenocrysts, similar to flows of the Imnaha Basalt of the Columbia River Basalt Group.

Flows of latite and quartz latite overlie the lower basalt and in places directly overlie the Silver City granite (Thomason, 1983). The latite and quartz latite unit has a maximum thickness of about 549 meters (Panze, 1975).

The Silver City rhyolite (Asher, 1968) forms much of the central core of the Owyhee Mountains (Ekren et al., 1984) and consists of numerous individual and coalesced rhyolite flows and domes derived from local eruptive centers, as well as intercalated units of rhyolite ash-flow tuff (Panze, 1971; 1975; Thomason, 1983). Thomason (1983) estimated a composite thickness of as much as 1,500 feet for the sequence. Panze (1975) recognized a consistent succession of quartz latite, flow breccia and upper rhyolite that can be traced through the central Owyhee Mountains, and defined several vent areas and individual domes. More recent studies have shown that some of the individual quartz latite and rhyolite units consist of flow-layered, rheomorphic ash-flow tuffs of regional extent (Ekren et al., 1984).

The western and southern flanks of the Owyhee Mountains are capped by one or more cooling units of the Swisher Mountain Tuff, which overlies the Silver City rhyolite (Figure 7.2; Thomason, 1983; Ekren et al., 1984). To the west of DeLamar, the Swisher Mountain Tuff was emplaced at about 13.8 Ma as a regional sheet of unusually high-temperature rhyolite ash flows erupted from a vent area located near Juniper Mountain, about 64 kilometers south of De Lamar and Silver City (Ekren et al., 1984). Most of the unit is extremely densely welded and underwent post-compaction internal flowage (rheomorphic deformation), resulting in brecciated vitrophyres, contorted flow laminations and internal flow brecciation. In some places, however, eutaxitic textures and preserved pumice clasts provide evidence for the original ash-flow emplacement (Ekren et al., 1984).

Map patterns indicate the Owyhee Mountains have undergone incipient to minor amounts of mid-Miocene and younger regional extension. The principal faults recognized in the central Owyhee Mountains have normal displacements and primarily north-northwest orientations (Figure 7.2) approximately parallel to the Northern Nevada Rift (Figure 7.1). As stated by Bonnichsen (1983), "*The attitude of the volcanic units generally ranges from subhorizontal to gently dipping, most commonly southwards. It is not clear if all the dips are due to initial deposition on uneven topography, or if some of the units have been rotated.*"

7.3 DeLamar Project Area Geology

7.3.1 DeLamar Area

Earth Resources and NERCO geologists defined a local volcanic stratigraphic sequence in the DeLamar area based on geologic mapping and drilling. Mapping at various times benefited from exposures in the

walls of the Glen Silver, Sommercamp – Regan, and North DeLamar pits. In addition to internal company reports, the geology of the DeLamar area has been documented in studies by Thomason (1983), Halsor (1983), Halsor et al. (1988), and Cupp (1989). These workers were involved with the exploration and operation of the project. The most concise and complete description of the local stratigraphic units and the mine area geologic setting was given by Halsor et al. (1988) and is presented here in Table 7.1. The Silver City granite is not exposed in the DeLamar area and has not been penetrated by drilling, although it is considered likely to underlie the Miocene rocks at depth.

Table 7.1 Summary of Volcanic Rock Units in the Vicinity of the DeLamar Mine
(modified from Halsor et al., 1988)

	Unit and symbol	Thickness (ft)	Phenocryst and rock fragment data	Description	Mode of emplacement and possible source	Comments
Silver City rhyolite	Millsite rhyolite (Tms)	0 to 500+	5 percent subhedral sanidine and quartz phenocrysts up to 3 mm across	Purplish red; flow breccias are common at top and base; massive to flow-banded interior with columnar joints; lithophysae are common	Lava flow(s) from north-northwest-trending dikes at Louse Mountain (sec 22, T 5 S, R 4 W)	Postmineralization only minor alteration
	Banded rhyolite (Tbr)	0 to 300	Less than 1 percent phenocrysts of rounded sanidine and quartz	White, pink, or purplish red; strongly developed folded flow bands; commonly pervasively altered; hydrothermally brecciated and veined by quartz	Low-viscosity lava flow or possibly an ash flow, probably from a local vent	50 to 70 ft of basal vitrophyre was altered to form a clay layer that ponded hydrothermal solutions
	Porphyritic rhyolite (Tpr)	100 to 850	3 to 8 percent subhedral to euhedral quartz and sanidine phenocrysts	Buff to white; generally homogeneous and massive, seldom banded; commonly silicified with quartz veins and brecciation in altered zones	Rhyolite dome or thick lava flow lobe from a nearby buried source; it may cover its own vent	One of many rhyolite domes in the region associated with north-northwest-trending faulting
	Tuff breccia (Ttb)	0 to 170	Angular fragments of altered Tlb and Tl up to 10 cm across in a fine, altered matrix	Green; predominantly bedded lapilli tuff; beds vary from several inches to several feet thick and are moderately sorted by size but are not graded	Near-vent outfall from phreatomagmatic explosions that probably culminated in Tpr extrusion	Unit is not laterally continuous; pervasive alteration; some fragments replaced by pyrite
	Quartz latite (Tql)	0 to 350(?)	Sparse phenocrysts of quartz, sanidine, and minor andesine and clinopyroxene less than 1 mm across	Black to greenish gray; weathers to orange or red; commonly altered to red or white; produces platy fragments and extensive talus on slopes	Lava flows mainly from Florida Mountain and Cinnabar Mountain and other sources	Unit of regional extent exposed in the Glen Silver pit and nearby in Louse Creek
	Porphyritic latite (Tl)	50 to 200	Xenocrysts and xenoliths of quartz; feldspar, granite, and basalt; 3 percent 1-mm-size quartz and feldspar phenocrysts	Dark gray to black; weathers to brownish red where massive and to various colors where platy; commonly altered to red or white; commonly has platy structure and red amygdules	Mainly vesicular lava flows from Sullivan Knob near DeLamar and Florida Mountains	Occurs above Tlb at the DeLamar silver mine; regional studies indicate Tl is intercalated within upper part of Tlb elsewhere
	Lower basalt (Tlb)	0 to 2,500+	Commonly has labradorite laths up to 1 cm long; local olivine phenocrysts	Black to gray-green; generally massive but locally scoriaceous, brecciated, palagonitic, and pillowed; commonly has poorly developed columnar jointing	Numerous lava flows; probably fissure eruptions from north-northwest-trending dikes	Flows typically are 50 to 150 ft thick and are quite continuous laterally

The mine geologists considered the units above the lower basalt to be subunits of the Silver City rhyolite. However, the quartz latite (unit Tql, Table 7.1) has been correlated with the tuff of Flint Creek, a regional, high-temperature lava-like ash-flow tuff (Ekren et al., 1984).

Figure 7.3 shows the principal mineralized zones of the DeLamar project in relation to the DeLamar project outline, Figure 7.4 shows the surface geology of these mineralized zones, and Figure 7.5 shows a schematic geological cross section. Open-pits of the DeLamar mine were developed at the Glen Silver, Sommercamp – Regan, and North DeLamar zones. The Sullivan Gulch and Milestone zones have not been mined.

Figure 7.3 Land Position Map Showing Mineralized Zones

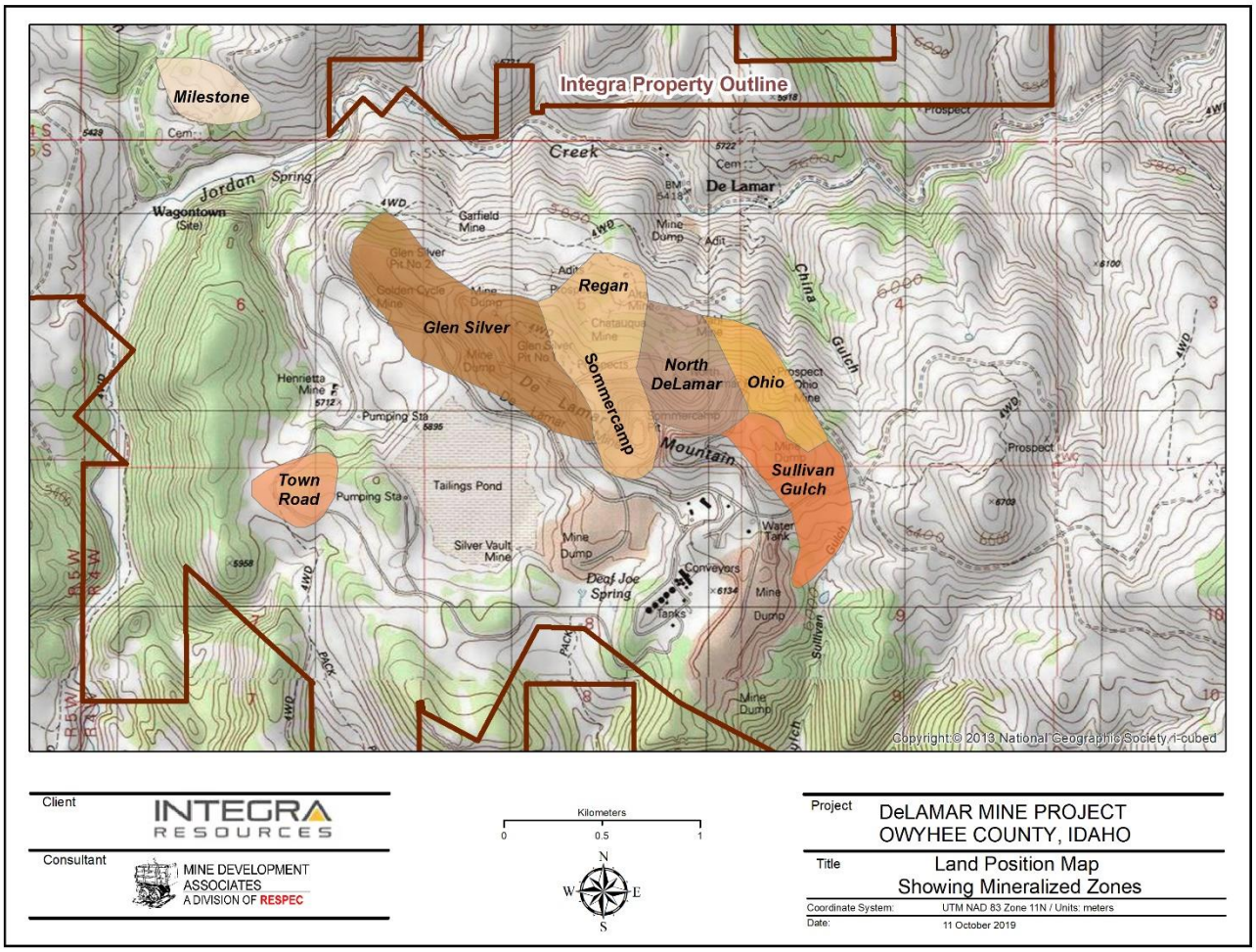
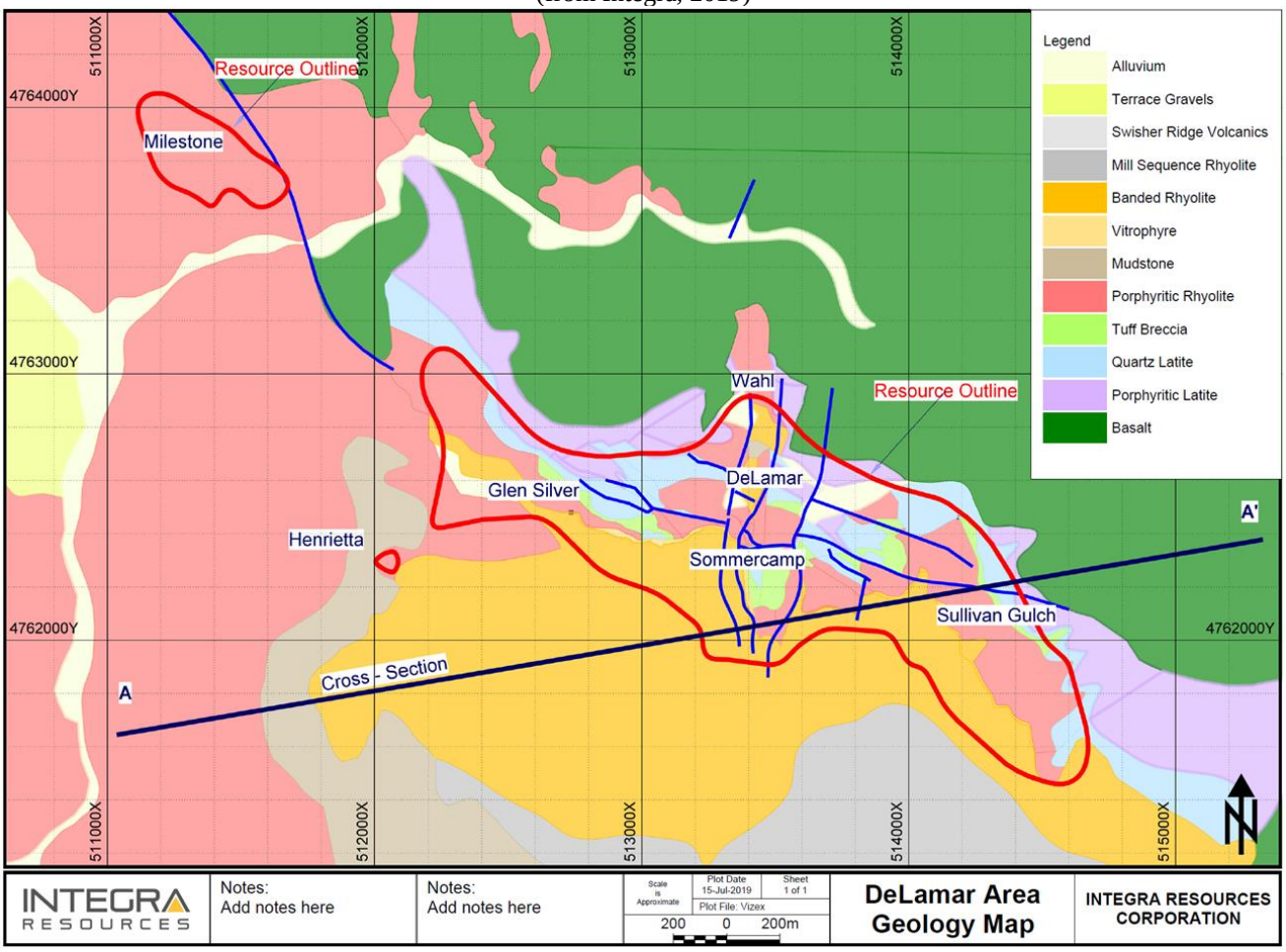


Figure 7.4 Integra Generalized 2018 DeLamar Area Geology

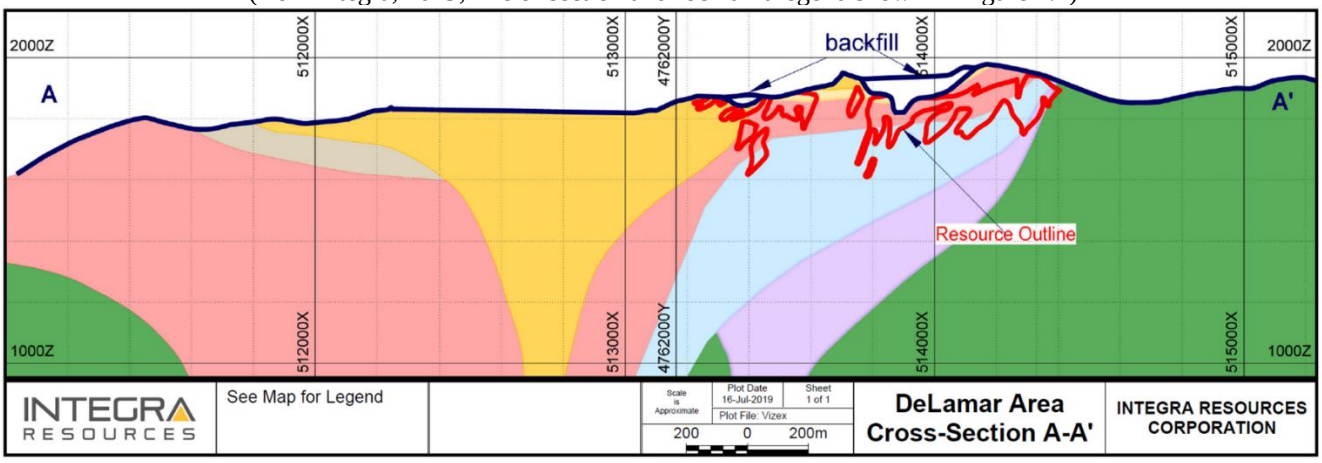
(from Integra, 2019)



Note: Red outlines are schematic surface projection of the resource footprint; blue lines are faults. UTM grid NAD83, Zone 11; Y = North, X = East

Figure 7.5 Integra 2018 Schematic Cross-Section, DeLamar Area

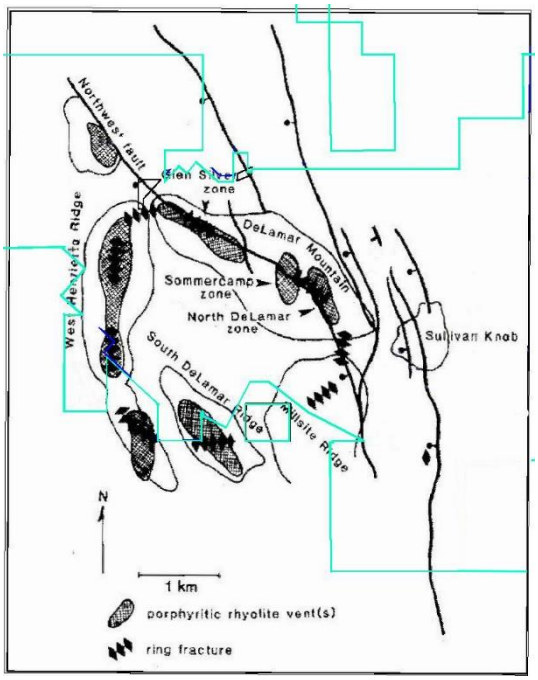
(from Integra, 2019; line of section and rock unit legend shown in Figure 7.4)



Note: see Figure 7.4 for geology legend. UTM grid NAD83, Zone 11; X = East, Y = North, Z = elevation in meters.

Mapping and drilling by Earth Resources and NERCO geologists has led to the interpretation that the mine area and mineralized zones are situated within an arcuate, nearly circular array of overlapping porphyritic and banded rhyolite flows and domes. These flows and domes overlie cogenetic, precursor pyroclastic deposits erupted as local tuff rings (Halsor, 1983; Halsor et al., 1988). Halsor (1983) interpreted the porphyritic and banded rhyolite flows and domes to have been emplaced along a system of ring fractures developed above a shallow magma chamber that supplied the erupted rhyolites, while Integra believes the rhyolites and latites were emplaced along northwest-trending structures as composite flow domes. The magma chamber was inferred to have been intruded within a northwest flexure of regional north-northwest trending Basin and Range faults (Figure 7.6).

Figure 7.6 Volcano-Tectonic Setting of the DeLamar Area
(showing land boundaries; modified from Halsor et al., 1988)



Core drilling in 2018 by Integra has facilitated the recognition of a unit of hydrothermally altered tuffaceous mudstone that is locally present between the porphyritic rhyolite and the overlying banded rhyolite as shown in Figure 7.5. This mudstone unit is up to 14 meters in thickness, strongly altered to clay, and includes fragmental volcanic layers of probable pyroclastic origin (Sillitoe, 2018; Hedenquist, 2018).

7.3.2 Florida Mountain – Stone Cabin Mine Area

The geology of the Florida Mountain area has been described in general by Lindgren (1900) and Piper and Laney (1926). More detailed studies were carried out by Earth Resources and NERCO as documented by Lindberg (1985), Porterfield and Moss (1988), and summarized by Mosser (1992). The oldest stratigraphic unit is the Late Cretaceous Silver City granite, which is unconformably overlain by the mid-Miocene lower basalt to trachyandesite lavas. The granite and lower basalt are overlain by a

sequence of andesitic volcanic-sedimentary and tuffaceous lacustrine rocks, which are in turn intruded and overlain successively by units of quartz latite, tuff breccia, and porphyritic rhyolite of the Silver City rhyolite (e.g., Lindberg, 1985). As at DeLamar, the tuff-breccia unit is interpreted as a near-vent pyroclastic unit erupted as a precursor to emplacement of the rhyolite flows and domes. NERCO's geologic map of the upper part of Florida Mountain is shown in Figure 7.7 and the explanation of map units is shown in Figure 7.8.

In contrast to the DeLamar area, the Silver City granite crops out on the flanks of Florida Mountain and was extensively penetrated by workings of the historical underground mines. It was designated granodiorite (unit Kgd of Figure 7.7) by the NERCO geologists. Field relations demonstrate the lower basalt flows partially buried an erosional, paleotopographic high of Silver City granite. Surface exposures and maps of the underground workings, as well as early drilling at Florida Mountain, led Lindberg (1985) to infer the granite forms a northeast-trending ridge beneath a relatively thin capping of quartz latite, tuff breccia, and one or more flows of rhyolite lava. Lindberg's schematic cross section through Florida Mountain is shown in Figure 7.9.

The Earth Resources and NERCO geologists interpreted certain rocks at Florida Mountain to represent volcanic vents from which portions of the rhyolite flows and possibly tuffs were presumably erupted (map units Thbx, Tpfv-bx, and Tfv of Figure 7.7 and Figure 7.8), and which later were important foci of hydrothermal activity, alteration, and mineralization (e.g., Porterfield and Moss, 1988; Mosser, 1992). However, exposures of rock units at Florida Mountain were generally poor prior to the start of mining by Kinross in 1994 as explained by Lindberg (1985), and the criteria used by the above authors to define the vent facies units and to delineate their geometries are not known to the authors. Moreover, most of the drilling at Florida Mountain was done by conventional rotary and RC methods, which can make outcrop-scale rock textural characteristics much more difficult, to impossible, to discern and correctly interpret.

Figure 7.7 Geologic Map of Florida Mountain
(from Porterfield and Moss, 1988)

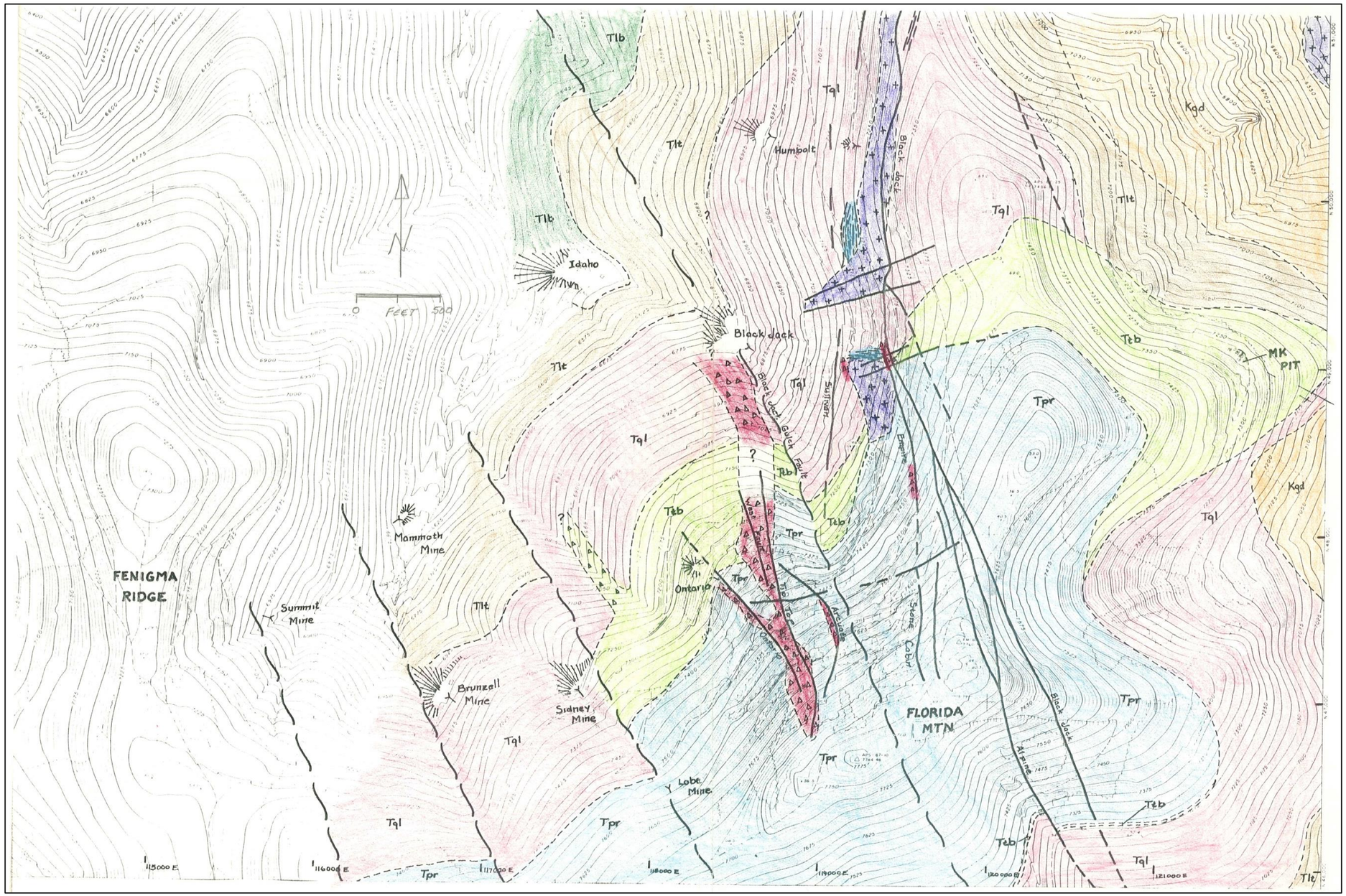


Figure 7.8 Map Legend for Florida Mountain Geology

(from Porterfield and Moss, 1988)

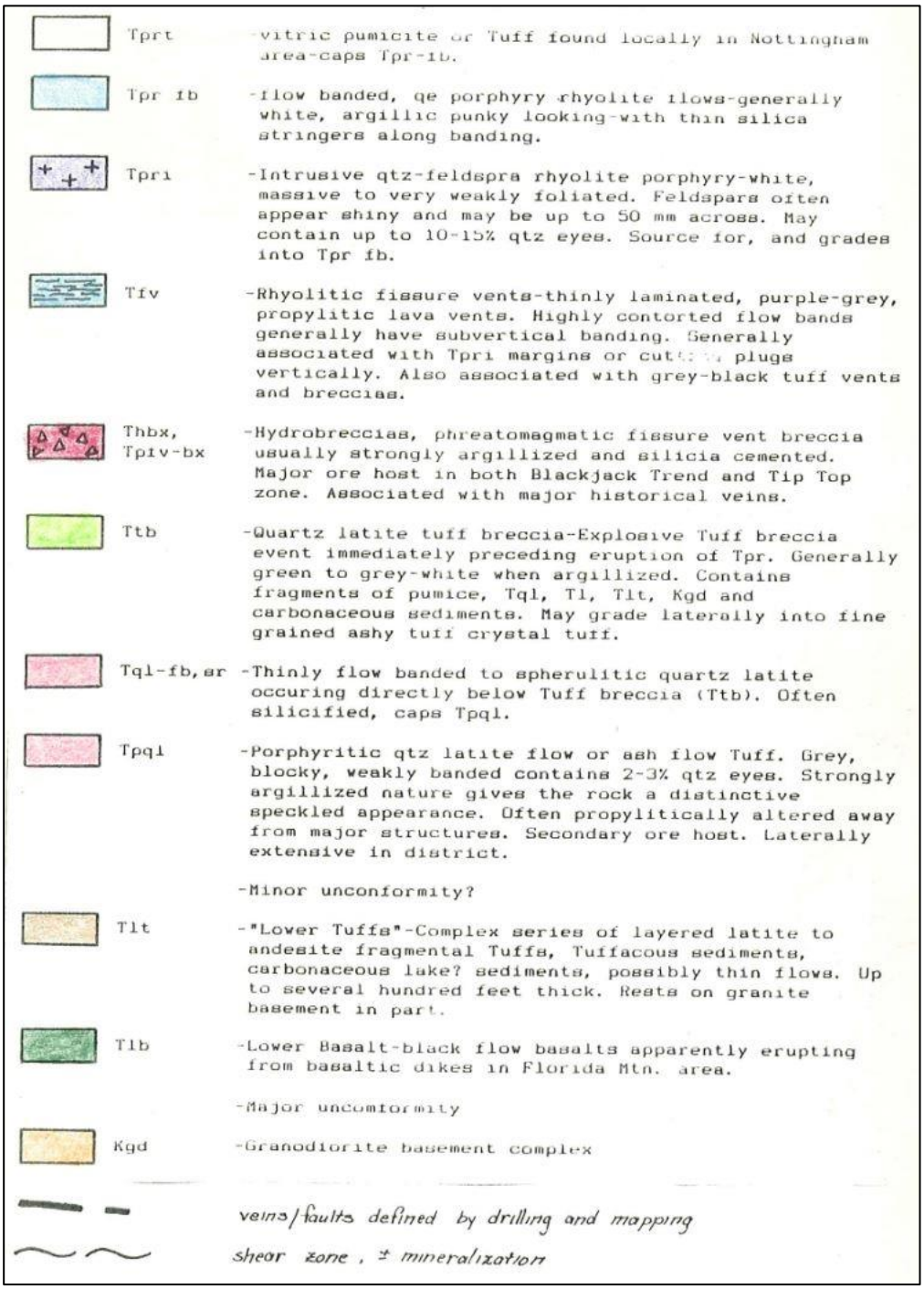
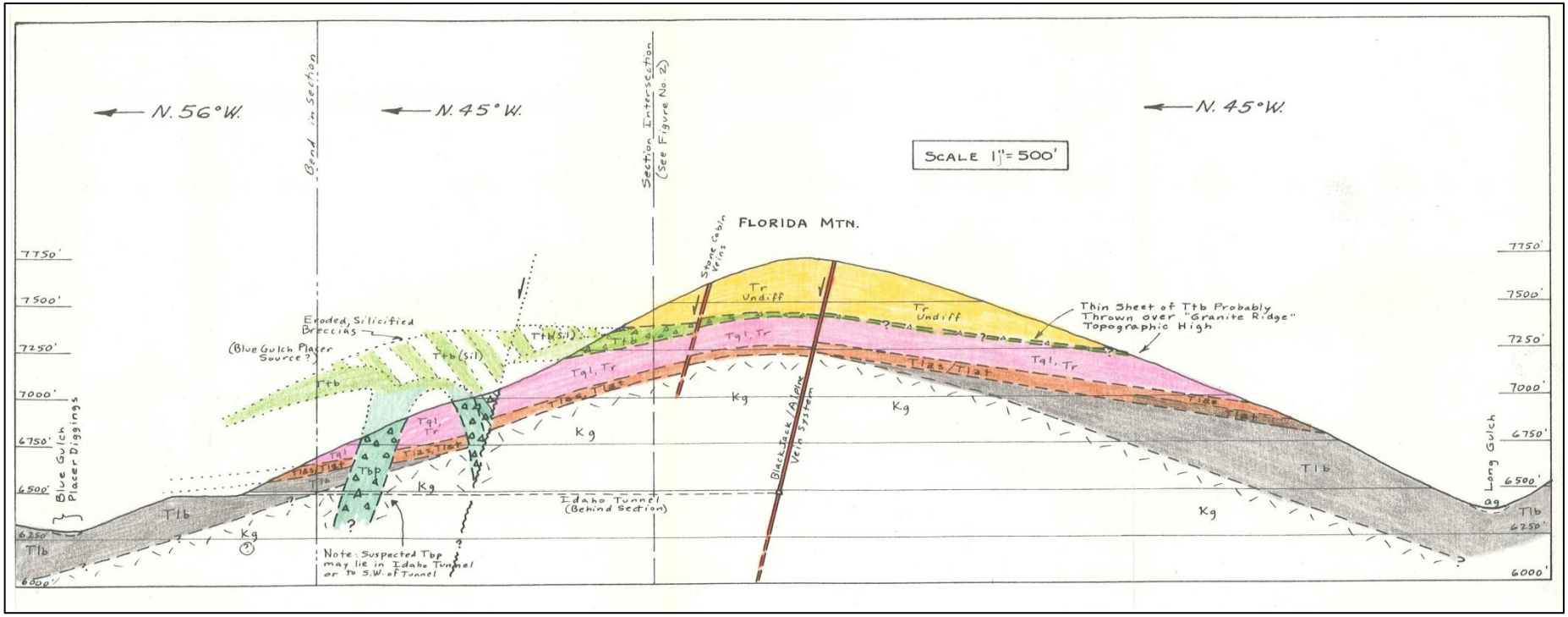


Figure 7.9 Schematic Florida Mountain Cross Section (Looking Northeast)

(from Lindberg, 1985)



Note: Kg = Cretaceous Silver City granite; Tlb = mid-Miocene lower basalt; Tlat and Tlas = show volcanic-sedimentary and tuffaceous lacustrine sequence; Tql, Tr shows quartz latite and ash-flow tuff (tuff of Flint Creek?); Ttb = tuff breccia; Tbp = tuff breccia pipe; Tr undiff. = rhyolite flows. Elevations in feet above sea level.

7.4 Mineralization

Numerous studies of the gold and silver mineralization in the DeLamar project - Silver City area have been conducted, beginning in the late 1860s. The most definitive studies and descriptions have been those of Lindgren (1900), Piper and Laney (1926), Thomason (1983), Halsor (1983), Halsor et al. (1988), and Mosser (1992). Mr. Gustin and Mr. Weiss have reviewed this information and believe it reasonably describes the mineralization as presently understood.

7.4.1 District Mineralization

Precious-metal mineralization has been recognized in two types of deposits: within 1) relatively continuous, quartz-filled fissure veins, and 2) broader, bulk-mineable zones of closely-spaced quartz veinlets and quartz-cemented hydrothermal breccia veinlets that are individually continuous for only a few feet laterally and vertically, and of mainly less than 1.3 centimeters in width.

Fissure Vein Mineralization

Mineralization mined from bedrock prior to 1942 was of the fissure vein deposit type. A concise summary of this type of mineralization in the Carson district was given by Bonnichsen (1983), as follows:

> *"Nearly all of the gold- and silver-bearing veins in the district strike north to northwest, following the main fault and dike trends, and are thought to be the same age....*
>
> *Most of the veins are fissures filled with quartz, accompanied by variable amounts of adularia, sericite, or clay. A few have been described as silicified shear zones."*

At the De Lamar underground mine, the veins were as much as 23 meters in width, but more commonly were 6 meters in width or less. Referring to veins in the Florida Mountain area, Bonnichsen (1983) went on to state:

> *"The veins are narrow, in most places only a few inches to a few feet wide, but persist laterally and vertically for as much as several thousand feet. Within an individual vein, the gold and silver ore occurs in definite shoots, generally with a moderate rake and somewhat irregular outline. The localization of ore shoots has commonly been attributed to the presence of cross-fractures, or, in one instance (Trade Dollar Mine), to the intersection of the vein with the granite-basalt contact. Some of the most productive veins in the district follow thin basaltic dikes.*
>
> *All three major rock units, the Silver City granite, the lower basalt-latite unit, and the Silver City rhyolite, are cut by mineralized veins. Most of the production at War Eagle Mountain, Florida Mountain, and Flint was from veins in the granite, while at De Lamar all of the production was from the rhyolite.*
>
> *Naumannite (Ag_2Se) is the principal hypogene silver mineral and normally is accompanied by variable but subordinate amounts of aguilarite (Ag_4SeS), argentite, and ruby silver as well as other silver-bearing sulfantimonides and sulfarsenides. Where interpreted to have been reorganized by supergene activity (Lindgren, 1900; Piper and Laney, 1926), the principal silver minerals are native silver, cerargyrite, and some secondary naumannite and acanthite. In both the hypogene and the oxidized and supergene-enriched portions of the veins, the principal gold-*

bearing minerals are native gold and electrum. Variable amounts of pyrite and marcasite, and minor chalcopyrite, sphalerite, and galena occur in some veins; the base metal-bearing minerals become more abundant at deeper levels.

Quartz is the principal gangue mineral. Much is massive, but some has drusy or comb structure and a lamellar variety is locally abundant. This lamellar (or cellular or pseudomorphic) variety consists of thin plates of quartz set at various angles to one another (see photographs in Lindgren, 1900; Piper and Laney, 1926). Each plate consists of numerous tiny crystals that have grown from either side of a medial plane. Lamellar quartz has been interpreted as the replacement of preexisting calcite (or perhaps barite) crystals. Adularia commonly shows crystal outlines developed as open-space fillings."

Calcite is reported to be present in only a few veins in the district, such as the Banner vein at Florida Mountain (Piper and Laney, 1926). Adularia is sparse in veins of the historical De Lamar mine, but is an abundant component of veins at Florida Mountain and War Eagle Mountain (Lindgren, 1900; Piper and Laney, 1926).

Potassium-argon age dates of volcanic units cut by veins, and dates on vein adularia concentrates, indicate that vein mineralization in the Silver City district was coeval with rhyolite volcanism at about 16 to 15 Ma (e.g., Panze, 1972; 1975; Halsor et al., 1988). More recent high-precision Ar^{40}/Ar^{39} ages of adularia extracted from four samples of veins immediately outside of the project range from 15.42 ±0.07 Ma to 15.58 ±0.06 Ma (Aseto, 2012), in good agreement with the earlier studies.

Bulk-Mineable Mineralization

Zones of bulk-mineable mineralization have been recognized in the district only since the early 1970s. Mining of this type of mineralization has only occurred in the DeLamar project at both the DeLamar and Florida Mountain areas. Accordingly, this type of mineralization is described below in Section 7.5.1 and Section 7.5.2.

7.5 DeLamar Project Mineralization

Current mineral resources discussed in this report are in the Florida Mountain area and the DeLamar area, which includes the Milestone prospect.

7.5.1 DeLamar Area

The modern DeLamar open-pit mine area encompasses the historical De Lamar mine where fissure-vein mineralization was mined from 1889 through 1913. Mineralized shoots in two sets of fissure veins, the Main De Lamar and Sommercamp veins, were mined from what are now the Sommercamp – Regan and North DeLamar open-pit zones of Figure 7.4, as shown in Figure 7.10 at the 4th level (elevation 1,902 meters).

Figure 7.10 Veins of the Historical De Lamar Mine, Elevation 6,240 Feet
(from Asher, 1968; based on Piper and Laney, 1926)

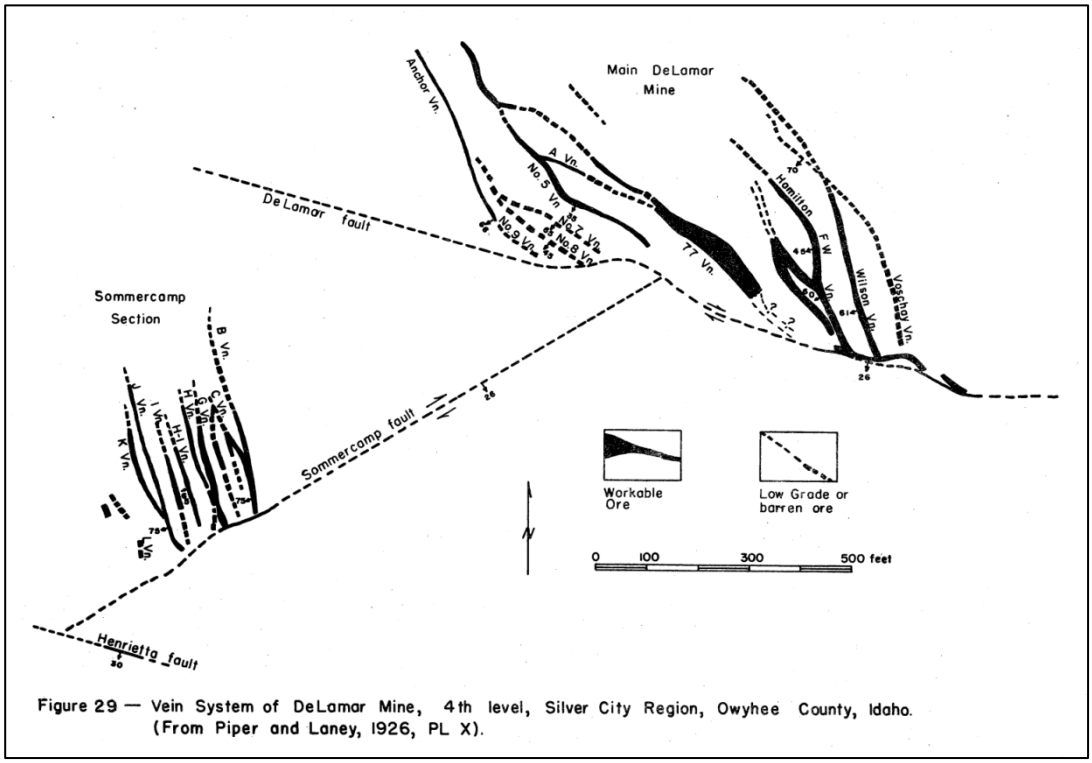


Figure 29 — Vein System of DeLamar Mine, 4th level, Silver City Region, Owyhee County, Idaho. (From Piper and Laney, 1926, PL X).

Note: the area of the above figure is entirely within the property boundary shown in Figure 7.3.

Bonnichsen's (1983) summary of the DeLamar area vein mineralization is as follows:

> *"The main De Lamar section, at the site of the present-day North DeLamar pit…was 1,300 feet long in a northwest-southeast direction and up to about 300 feet wide, as measured on the No. 4 level (6,240 feet elevation). The section contained the Hamilton-Wilson No. 9 vein striking N. 25° W. and dipping 45°-66° W., and the 77 vein striking N. 62° W. and dipping 35° SW. These were connected by smaller veins and stringers. At lower levels the veins assumed steeper dips, 65 to 80 degrees being common. The 77 vein was the most important producer. The Sommercamp section, at the site of the present-day Sommercamp pit…was a zone about 300 feet across that contained ten interlinked veins striking N. 18° W. and dipping 65°-80° W.*
>
> *These ore-bearing zones plunged 20 to 30 degrees southward. In both, the southern limit of the ore was a clay zone several feet thick with a shallow dip to the south. These clay zones were known as iron dikes to the miners and were interpreted to be the low-angle De Lamar and Sommercamp faults by Piper and Laney (1926), Asher (1968), and Panze (1975). However, the excellent exposure in the present-day open-pit mines has shown that these zones really are mainly the thick basal vitrophyric section of the banded rhyolite unit (Tbr) which has been hydrothermally altered. In the underground workings, much of the rich silver ore—the "silver talc"—was extracted where the veins abutted against the base of this clay zone. With its shallow dip, this zone formed the upper as well as the southern limit to mineralization in both sections of the mine."*

An indication of the grades mined can be found in Piper and Laney (1926), where the 77 vein was reported to have been stoped from 1893 through 1908 with average grades mainly of 17.14 – 20.57 grams gold per tonne, and about 44.57 – 1,714 grams silver per tonne, over widths of 0.305 to 7.3 meters. The overall width of the 77 vein was as much as 23 meters. During this period most of the production came from elevations above 1,786 meters, but some stopes were as deep as the 12th level at 1,768 meters. Although the 77 vein was found to persist to the 16th level at an elevation of 1,712 meters, the lowest elevation of workings, grades were largely sub-economic below the 10th level and only a small amount of production came from the 12th level (Piper and Laney, 1926). As pointed out by Piper and Laney (1926), there was little underground exploration, and the development that was done did not consider the southerly plunge of mineralization.

In addition to the fissure veins, the bulk mineable type of mineralization has been delineated in four broad, lower-grade zones, two of which overlap and are centered on the Sommercamp and main De Lamar fissure veins. This type of mineralization has been described by Halsor et al. (1988) as follows:

> *"Low grade mineralization occurs in porphyritic rhyolite where closely spaced veinlets and fracture fillings provide bulk tonnage ore. Most of the veinlets are less than 5 mm in width and have short lengths that are laterally and vertically discontinuous....Locally, small veins can form pods or irregular zones up to 1 to 2 cm wide that persist for several centimeters before pinching down to more restricted widths. In highly silicified zones, porphyritic rhyolite is commonly permeated by anastomosing microveinlets typically less than 0.5 mm wide. Most of the minute veining displays well-defined contacts with the enclosing rock and in some instances veins can be seen to sharply cut phenocrysts. Still, in other zones, microveinlets are less distinct and difficult to distinguish from groundmass silicification.*
>
> *Networks of high-density, quartz-free fractures are the sites for supergene mineralization. Major fractures generally trend north-northwest, but less prominent intervening and crosscutting fractures are present. Major fractures commonly have steep dips and show reversals in direction of dip vertically along faces. Fracture fillings commonly consist of thin coatings of goethite and jarosite but occasionally can be filled with seams of sericite and kaolinite up to several centimeters wide. Above the clay zone, veining is characterized by narrow, chalcedony-lined fractures of irregular extent.*
>
> *In the Sommercamp pit, the principal ore zone in porphyritic rhyolite occurred beneath the clay zone as a distinct shoot striking north-northwest, dipping 40° E; and plunging 9½° SE. It was 27 m thick at the south end and thickened to 90 m at the north end. The ore-waste boundary at the base of the shoot was sharp with ore-grade material (>2 oz Ag) in the shoot abruptly dropping to waste across a single 1.5-m sample interval. The base of the ore shoot was remarkably planar but dipped 40° E as mentioned above. The top of the ore shoot was undulatory and more or less defined by the base of the clay zone over the porphyritic rhyolite. Generally, major mineralized shoots in the Glen Silver, North DeLamar, and Sullivan Gulch zones all plunge 10° to 15° to the southeast. Determining the plunge in the North DeLamar pit proved difficult due to a very complex cross faulting pattern.*
>
> *Ore mineralogy is reported by Thomason (1983) and Barrett (1985). Naumannite (Ag_2Se) is the dominant silver mineral and acanthite (Ag_2S) and acanthite-aguilarite [(Ag_2S)-$(Ag_4)(Se,S)_2$] solid solution are the second most abundant. Remaining ore minerals consist of lesser amounts of argentopyrite ($AgFe_2S_3$), Se-bearing pyrargyrite [$Ag_3Sb(S,Se)_3$], Se-bearing polybasite*

[$(Ag,Cu)_{16}Sb_2(S,Se)_{11}$], cerargyrite [AgCI], Se-bearing stephanite [$Ag_5Sb(S,Se)_4$], native silver, and native gold and minor Se-bearing billingsleyite [$Ag_7(Sb,As)(S,Se)_6$], pyrostilpnite [$Ag_3Sb(S,Se)_3$] and Se-bearing pearceite [$(Ag,Cu)_{16}As_2(S,Se)_{11}$]. Ore minerals are generally very fine grained; 65 percent of the minerals average 62μ in diameter, with the remainder averaging 200μ (Rodgers, 1980). Naumannite, the dominant silver mineral, commonly occurs as finely disseminated grains in quartz veinlets and within some fractures. It is also found as crystal aggregates growing on drusy quartz that lines vugs. Acanthite, the second most abundant silver mineral, occurs as anhedral blebs in quartz gangue and hydrothermal clays commonly associated with naumannite. It also is frequently present as a late-stage mineral coating drusy quartz in vugs.... Pyrite is the most widespread metallic mineral occurring in veins and altered country rock. Pyrite occurs along the edges of veins but also as coatings on some of the younger minerals. Polymorphic marcasite is commonly associated with pyrite, forming lath shaped crystals and anhedral aggregates surrounding pyrite. In some zones, marcasite is intimately intergrown in irregular clots with pyrite....

Vein gangue minerals consist almost entirely of quartz, with minor amounts of mosaic intergrowths of adularia. Texturally, quartz can be divided into three varieties: (1) cloudy, massive, fine-grained quartz, (2) lamellar quartz, and (3) clear, crystalline, coarse-grained quartz.... Cloudy, fine grained quartz, including a chalcedonic variety, is the dominant type in veins and veinlets that constitute ore. This quartz is characterized by turbid anhedral grains (<0.005 mm) rich in solid inclusions.

The host rocks at DeLamar are pervasively altered. The tuff breccia is altered to an assemblage of quartz, illite, pyrite, and marcasite. The alteration of the principal host of mineralization, porphyritic rhyolite, is vertically zoned. The alteration assemblage is quartz, illite, pyrite, and marcasite and locally in the upper portions there are complex assemblages including jarosite, and mixtures of alunite, goethite, and kaolinite; hematite with kaolinite; and illite plus kaolinite (Thomason, 1983; Barrett, 1985). The latter style of alteration produces a very conspicuous glaring white rock that overlies the principal ore zones at DeLamar. The porphyritic rhyolite is overlain by a clay zone which consists of variable quantities of mixed layers of illite and montmorillonite clays with 5 to 7 vol percent euhedral pyrite in fine-grained aggregates or as crystals up to a few millimeters across. In less altered areas relic perlitic structure can be seen, demonstrating that the clay zone was a basal vitrophyre of the banded rhyolite. Above the clay zone, feldspar in the banded rhyolite is altered to kaolinite and the groundmass contains finely disseminated hematite, trace amounts of epidote, and patches of cryptocrystalline quartz. Sparse chemical data (Halsor, 1983) indicate that at least some of the DeLamar rocks were potassium metasomatized.

Scattered zones of breccia in the banded rhyolite occur most frequently near the base of the unit. These breccias crosscut flow layering, some ranging up to several meters in length by several decimeters in width. The breccias consist of close-packed angular fragments of flow-banded rhyolite in a chalcedonic matrix. The fragments show little rotation and this, together with the crosscutting nature of the breccias, suggests a hydrothermal origin and not primary features related to flow."

The above description seems to have been based on the Sommercamp and North DeLamar mineralized zones. Mr. Gustin and Mr. Weiss have no information to suggest that the Glen Silver and the unmined

Sullivan Gulch mineralization is different in a general sense. However, there is no indication that major fissure-vein mineralization was mined historically or encountered in exploration drilling in the Sullivan Gulch and Glen Silver zones, where relatively shallow drilling to date has intersected mineralization of the bulk mineable type.

Based on Integra's core drilling, the clay zone described above by Bonnichsen (1983) and Halsor et al. (1988) at least locally consists of the altered mudstone unit between the porphyritic and flow-banded rhyolites. The clay zone is interpreted as having acted as an important aquitard and barrier to upwelling hydrothermal fluids during mineralization (Sillitoe, 2018).

Samples from three drill-core intervals were studied with optical microscopy and x-ray powder diffraction methods at Hazen Research Inc. ("Hazen") in 1971 (Perry, 1971). In addition to identifying some of the silver minerals recognized by Thomason (1983) and Halsor (1988), the Hazen study noted that gold occurs as native gold and in electrum. The gold grains were reported to be "blebs" that "rarely exceed 5 microns in size" intergrown with quartz, and within and on naumannite (Perry, 1971). Electrum was found as silvery, nearly white blebs less than 5 microns in size "locked in cerargyrite".

The DeLamar area mineralization is situated stratigraphically below the Millsite rhyolite, which is reported to be little affected by hydrothermal alteration and is considered to be post-mineral in age (Thomason, 1983; Halsor et al., 1988).

7.5.1.1 Milestone Prospect

A shallow, hot-spring setting has been described by Barrett (1985) for gold-silver mineralization at the Milestone prospect, about 1 kilometer northwest and along the strike of the Glen Silver zone (Figure 7.3). According to Gierzycki (2004b):

> "*The ore lies at the base of a basalt-rhyolite contact in hydrothermal eruption-breccia with clasts of porphyritic rhyolite within a large zone of cherty silicification. It is capped at the surface by a sinter....Major ore minerals are naumannite, Se-rich pyrargyrite and gold."*

7.5.2 Florida Mountain Area

Both fissure veins and the bulk-mineable type of mineralization are present at Florida Mountain and both have contributed to past gold and silver production. The veins cropped out intermittently near the crest and on the flanks of Florida Mountain, in some cases with lateral continuity of 1.6 kilometers or more, even though vein widths were usually only a few meters or less. Dips are reported to be 75° to 80° W, transitioning in their northern extents to steep east dips (Piper and Laney, 1926). A longitudinal section showing stopes of the Black Jack – Trade Dollar mine is presented in Figure 7.11.

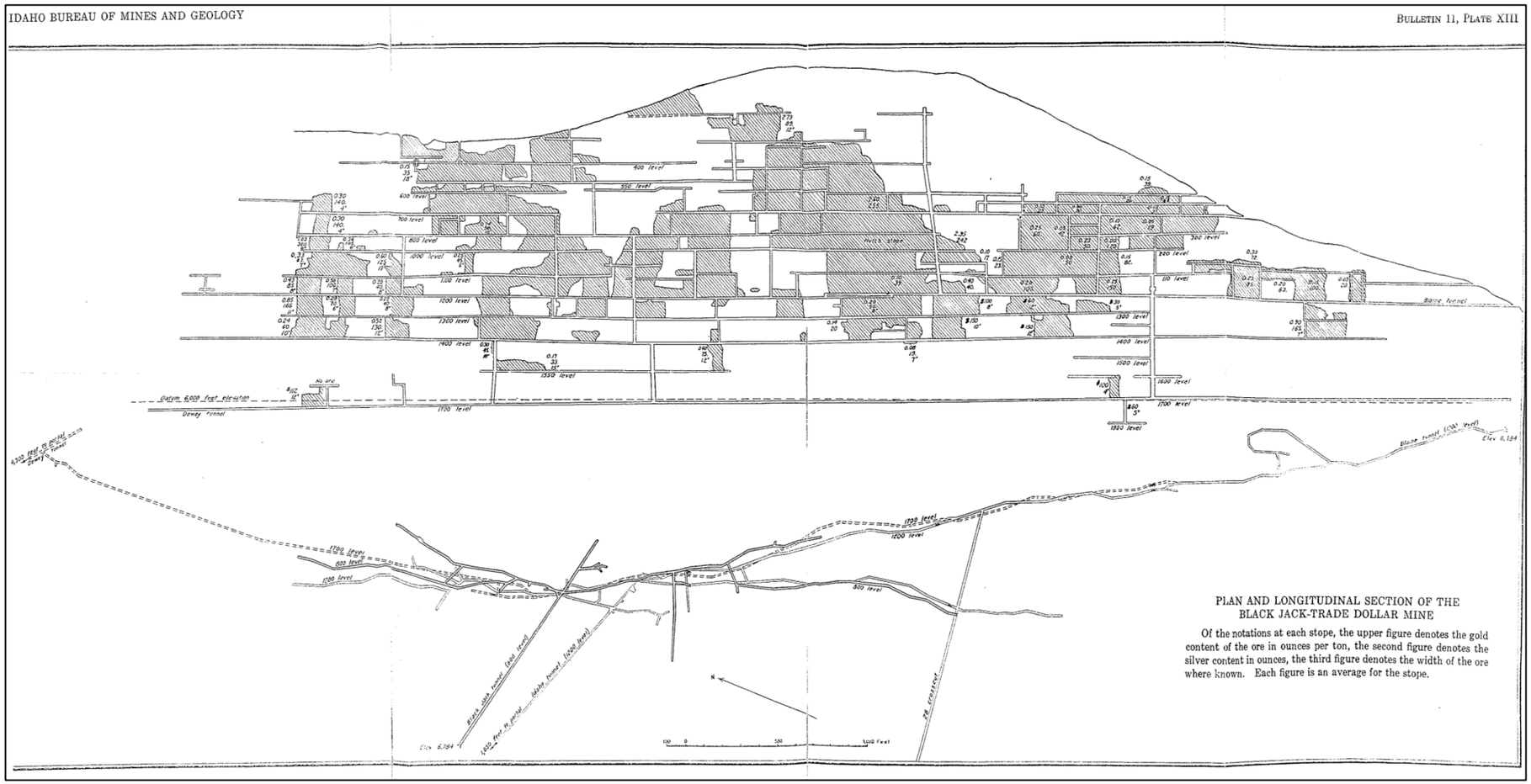
The veins in Florida Mountain were mapped in greater detail in the 1970s and 1980s by Earth Resources and NERCO geologists (e.g. Figure 7.7), in part with the benefit of trenching and drilling. The most complete vein and geologic map that Mr. Gustin and Mr. Weiss are aware of is a NERCO map from 1989. The NERCO 1989 map shows a somewhat different, more detailed picture of the vein array than Piper and Laney's 1926 map.

Mosser (1992) summarized the vein mineralization as follows:

> *"...Mineralization is strongly controlled by NNW-trending faults, and to a lesser degree by arcuate and ENE structures. Host rocks display a definite influence on mineral distribution. Within the granodiorite and basalt, where most of the historic production occurred, the veins are narrow and tight. However, within the more reactive and permeable quartz-latite and rhyolite units, the mineralization is more disseminated so that significant bulk mineable potential exists...*
>
> *The vein deposits are dominated by quartz and adularia gangue. Quartz occurs in a variety of forms in a definite paragenetic sequence....*
>
> *Hypogene gold and silver mineralization varies little with depth across known levels and is dominated by electrum, acanthite, and the silver sulfo-selenide aguilarite...."*

In the quartz latite and rhyolite, at least some of the veins branch upward into multiple narrow veins and vein-cemented breccia, separated by intensely altered rhyolite, to form sheeted vein and breccia zones as much as 6.1 meters or more in width. These broader sheeted vein and breccia zones comprise the bulk-mineable style of mineralization at Florida Mountain, particularly where adjacent fracture networks and flow bands in the rhyolite have been permeated with narrow, discontinuous quartz and breccia veinlets. Four such zones were described by Mosser (1992), referred to as the Tip Top, Stone Cabin, Main Trend (Black Jack), and Clark deposits. The mineralogy and paragenesis of the gold and silver mineralization are similar, if not the same, as that described for the fissure veins. Details of the mineralogy and a fluid inclusion study were presented by Mosser (1992).

Figure 7.11 Longitudinal Section of the Black Jack – Trade Dollar Mine
(from Piper and Laney, 1926)

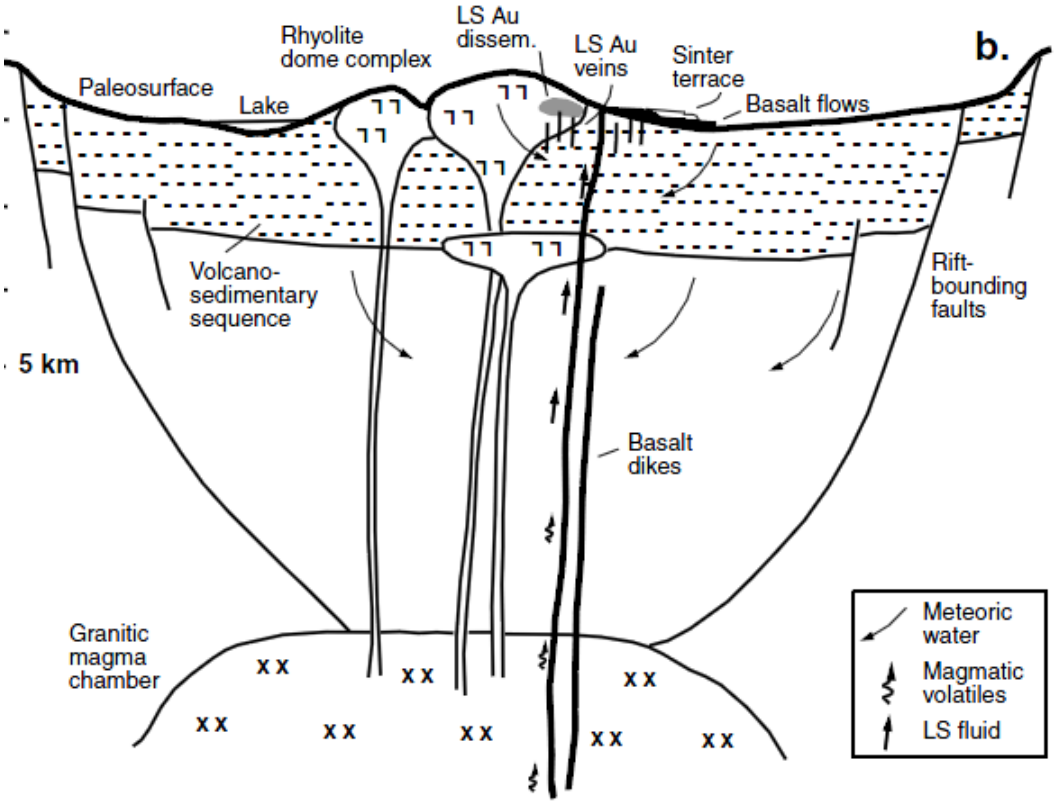


8.0 DEPOSIT TYPE

Based upon the styles of alteration, the nature of the veins, the alteration and vein mineralogy, and the geologic setting, the gold and silver mineralization at the DeLamar project is best interpreted in the context of the volcanic-hosted, low-sulfidation type of epithermal model. This model has its origins in the De Lamar - Silver City district, where it was first developed by Lindgren (1900) based on his first-hand studies of the veins and altered wallrocks in the De Lamar and Florida Mountain mines. Various vein textures, mineralization, and alteration features, and the low contents of base metals in the district are typical of what are now known as low-sulfidation epithermal deposits world-wide. Figure 8.1, below, from Sillitoe and Hedenquist (2003), is a conceptual cross-section depicting a low-sulfidation epithermal system. The host-rock setting of mineralization at the DeLamar project is similar to the simple model shown in Figure 8.1, with the lower basalt sequence occupying the stratigraphic position of the volcano-sedimentary rocks shown below. The Milestone portion of the district appears to be situated within and near the surficial sinter terrace in this model.

Figure 8.1 Schematic Model of a Low-Sulfidation Epithermal Mineralizing System

(After Sillitoe and Hedenquist, 2003)



As documented by Lindgren (1900) and Piper and Laney (1926), many of the veins in the district contain distinctive boxwork and lamellar textures where quartz has replaced earlier crystals of calcite. These textures are now known to result from episodic boiling of the hydrothermal fluids from which the veins were deposited. Limited fluid inclusion studies of quartz from veins in the upper part of Florida

Mountain by Mosser (1992) support the concept of fluid boiling and indicate fluid temperatures were in the range of 235°C to 275°C. Salinities measured by freezing point depressions were apparently in the range of 0.25 to 2.1 equivalent weight percent NaCl, with a mean of about 0.8 equivalent weight percent NaCl (Mosser, 1992). Halsor et al. (1988) reported fluid temperatures from late-stage quartz in the DeLamar mine of about 170°C to 240°C, with salinities of 2.8 to 3.8 equivalent weight percent NaCl. The temperature and salinity data, and evidence for fluid boiling are typical of the low-sulfidation epithermal class of precious-metal deposits world-wide.

Many other deposits of this class occur within the Basin and Range province of Nevada, and elsewhere in the world. Some well-known low-sulfidation epithermal gold and silver properties with geological similarities to the DeLamar project include the past-producing Rawhide, Sleeper, Midas, and Hog Ranch mines in Nevada. The Midas district includes selenium-rich veins similar to, but much richer in calcite, than the veins known in the DeLamar project. At both DeLamar and Midas, epithermal mineralization took place coeval with rhyolite volcanism, and shortly after basaltic volcanism, during middle Miocene time.

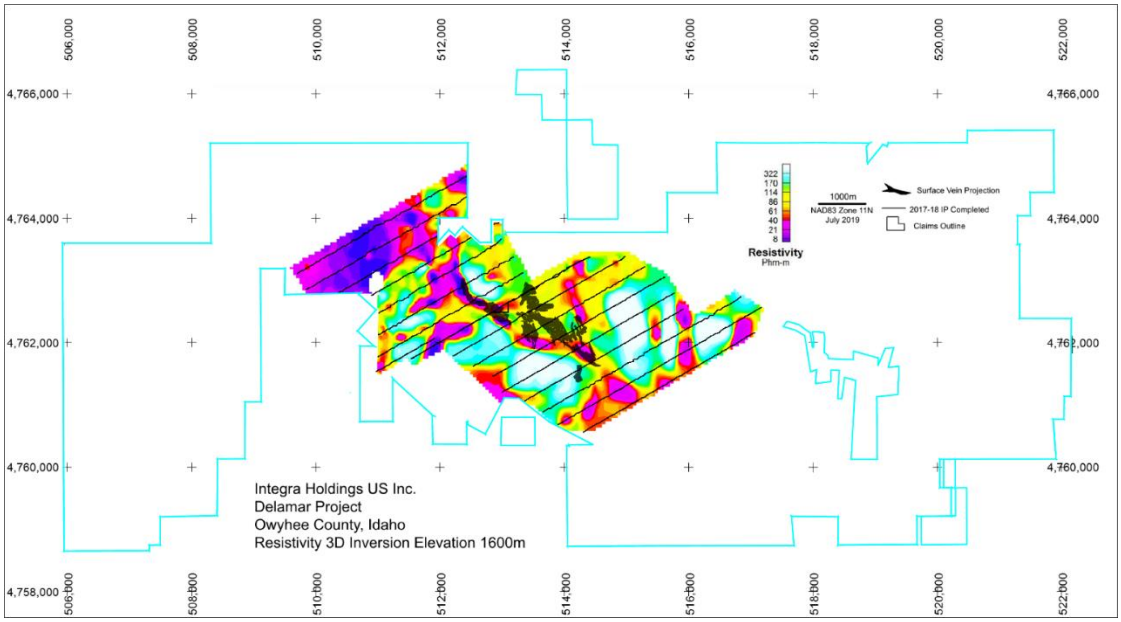
9.0 EXPLORATION

This section summarizes the exploration work carried out by Integra. Drilling by previous operators is summarized in Sections 10.2 and 10.3. Integra commenced drilling in 2018 on patented claims in the DeLamar area of the project and subsequently conducted drilling elsewhere at DeLamar as well as in the Florida Mountain area. Drilling conducted by Integra is described in Section 10.4 and was on-going as of the effective date of this report.

9.1 Topographic and Geophysical Surveys

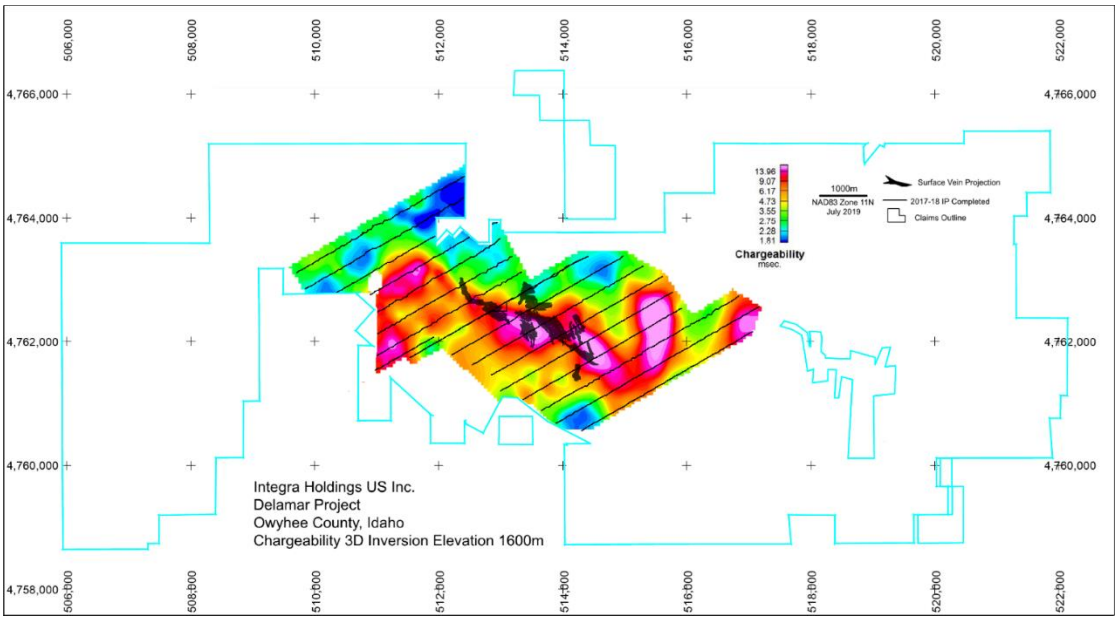
A Light Detection and Ranging ("LiDAR") topographic survey of the DeLamar and Florida Mountain areas was completed late in 2017. Integra also commissioned SJ Geophysics Ltd., of Delta, British Columbia, to conduct an Induced Potential and Resistivity ("IP/RES") survey of six lines using the Volterra-2DIP distributed array system for a total of 22.4 line-kilometers in the DeLamar area late in 2017. The survey was extended with an additional 10 lines in 2018, bringing the total survey to approximately 40 line-kilometers. The IP/RES lines were spaced at 300 meters and utilized a potential dipole spacing with intermediate current spacing of 100 meters. The results are shown in Figure 9.1 and Figure 9.2.

Figure 9.1 Plan View of Resistivity from 2017 and 2018 IP/RES Surveys
(from Integra, 2019; 3D inversion elevation 1,600 meters)



Note: heavy black lines for "Surface Vein Projection" are schematic representations of historically mined mineralized structures; north is up.

Figure 9.2 Plan View of Chargeability from 2017 and 2018 IP/RES Surveys
(from Integra, 2019; 3D inversion elevation 1,600 meters)



Note: heavy black lines for "Surface Vein Projection" are schematic representations of historically mined mineralized structures; north is up.

9.2 Rock and Soil Geochemical Sampling

Integra conducted rock-chip and soil geochemical sampling at the DeLamar area in 2018. A total of 2,920 soil samples in the DeLamar area were collected at 50-meter intervals along lines spaced 300 meters apart, and 475 rock-chip samples were also collected.

9.3 Database Development and Checking

A major effort in updating the DeLamar and Florida Mountain drill-hole databases was undertaken by Integra. Geologists re-logged cuttings from nearly 2,500 historical RC drill holes and the re-logging data was added to the databases. This program included logging of oxidation (oxidized, transitional and unoxidized (sulfide), the data for which had never before been collected. Mr. Gustin used this logging to create detailed oxidation models for both resource areas. In addition, Integra extracted information on underground workings, groundwater and/or moisture level of samples, sample quality, and notes of down-hole contamination from approximately 2,200 historical paper geologic logs stored at the project site and entered this information into electronic spreadsheets. MDA then augmented the project resource databases using these spreadsheets.

Integra also completed an extensive comparison of the DeLamar and Florida Mountain drilling assays to the original paper laboratory assay records. First, all drill-hole intervals which were missing assay data were identified. The historical paper laboratory assay records were then searched for the corresponding drill-hole intervals. In most cases, the gaps in assayed intervals were found to be "No Sample" intervals,

and the databases were updated with the No Sample designation if this was the case. If assays were found for these intervals, the data was added to the databases. Next, approximately every 10th sample interval in the databases was compared to the original paper records. This amounted to 7.5% of the Florida Mountain intervals and 9.7% of the DeLamar intervals. The drilled interval 'from' and 'to' depths, as well as the gold and silver assays for the interval, were compared to the paper records. The few discrepancies found were corrected with the entries recorded in the paper records and a field in each database was attributed with a record of the checking. This work was in addition to the verification work completed by the authors that is summarized in Section 12.0.

9.4 Cross-Sectional Geologic Model

Utilizing the updated databases, as well as available surface geology, Integra geologists constructed 100 hand-drawn cross-sections at 30-meter spacing through the DeLamar mine area. Cross-sectional lithology and structure were interpreted on each section using the down-hole data. Trends of mineralized zones were interpreted on each of the sections as well using the down-hole assays. While working on this modeling, conflicts in the geological coding of nearby holes were inevitably discovered. The resolution of these discrepancies often led Integra to update the lithologic codes in the project database with their own logging of the historical RC chips. Cross-sectional geological modeling was also completed at Florida Mountain prior to the updating of the databases described above.

10.0 DRILLING

The drilling described in this section was performed in the DeLamar and Florida Mountain areas of the property. This drilling was completed by historical operators from the late 1960s through 1998, and by Integra commencing in 2018.

10.1 Summary

MDA has records for a total of 306,078 meters drilled in 2,718 holes in the DeLamar and Florida Mountain portions of the property as summarized in Table 10.1. This includes Integra's drilling through April 2019, i.e. the drill holes used in the current resource estimation. Drilling at the project has continued through to the effective date of this report, but these new drill holes are not considered herein.

Table 10.1 DeLamar Project Drilling Summary

Area	Years	Holes	Meters
Historical Drilling			
DeLamar	1966 - 1998	1,447	136,097
	not known	103	6,693
Florida Mountain	1972 - 1997	1,060	131,228
	not known	15	1,772
Total Historical		*2,625*	*275,790*
Integra Drilling			
DeLamar	2018 - 2019	92	30,640
Florida Mountain	2018	10	2,932
Total Integra Drilling		*102*	*33,572*
Grand Total	***1966 - 2019***	***2,727***	***309,362***

Records of historical drilling are incomplete with respect to dates, drilling methods, drilling contractors, and types of drills used. As of the effective date of this report, MDA has documentation for 2,625 historical holes drilled in the DeLamar area, including the Milestone prospect, and the Florida Mountain area, for a total of 275,790 meters. Table 10.2 summarizes the historical drilling by operator and year.

Of the historical holes for which the drilling method is known, 602 of the DeLamar area holes were drilled by RC, 438 by conventional rotary, and 60 were core holes. Seventy-four percent of the historical holes in the DeLamar area were vertical. At Florida Mountain, 961 of the historical holes were drilled by RC methods, 58 by conventional-rotary methods, and 46 by diamond core methods; less than 10% of the historical holes were vertical. None of the conventional rotary holes were angled in either area. A combined total of 106 holes were drilled using core methods for a total of 10,822 meters, or 3.9% of the overall meterage drilled. The median down-hole depth of all historical holes in the DeLamar area is 91 meters, and the median depth in the Florida Mountain area is 123 meters. The aerial distribution of drill holes in the DeLamar area is shown in Figure 10.1. Historical drilling in the Florida Mountain area is shown in Figure 10.2.

Table 10.2 Historical Drilling at the DeLamar and Florida Mountain Areas

Year	Company	Holes	Meters
	DeLamar Area		
1966	Continental Materials	5	1,378
1969 - 1983	Earth Resources	504	44,346
1972	Sidney Mining	8	654
1985 - 1992	NERCO	691	68,354
1993 - 1998	Kinross	239	21,365
no known	not known	103	6,693
DELAMAR TOTALS		***1,550***	***142,790***
	Florida Mountain Area		
1972	Earth Resources	16	1,236
1975 - 1976	Earth Resources	29	2,169
1977	ASARCO	4	579
1980	Earth Resources	9	651
1986 - 1990	NERCO	898	116,217
1988	NERCO Water Wells	5	476
1995 - 1997	Kinross	99	9,901
not known	not known	15	1,772
FLORIDA MOUNTAIN TOTALS		***1,075***	***133,000***
TOTAL PROJECT DRILLING		***2,625***	***275,790***

10.2 Historical Drilling – DeLamar Area

10.2.1 Continental 1966

The earliest drilling that MDA is aware of was completed by Continental in the area of the 77 vein of the old De Lamar underground mine and now the site of the North DeLamar pit. A total of 1,378 meters were drilled in five inclined core holes, but MDA is unaware of what type of drill rig was used, core diameter(s), or the identity of the drilling contractor.

10.2.2 Earth Resources 1969 - 1970

In 1969 and 1970, Earth Resources drilled 39 conventional rotary holes, for a total of 2,303 meters, in the North DeLamar, Sommercamp, and Glen Silver areas. All of the holes were vertical. Harris Drilling was the contractor for most of the drilling, some of which was done with a Failing 1500 drill rig. Eklund Drilling of Elko, Nevada, drilled one of the holes using a Mayhew 2000 drill. MDA is unaware of the type(s) and size(s) of drill bits used.

Figure 10.1 Map of DeLamar Area Drill Holes

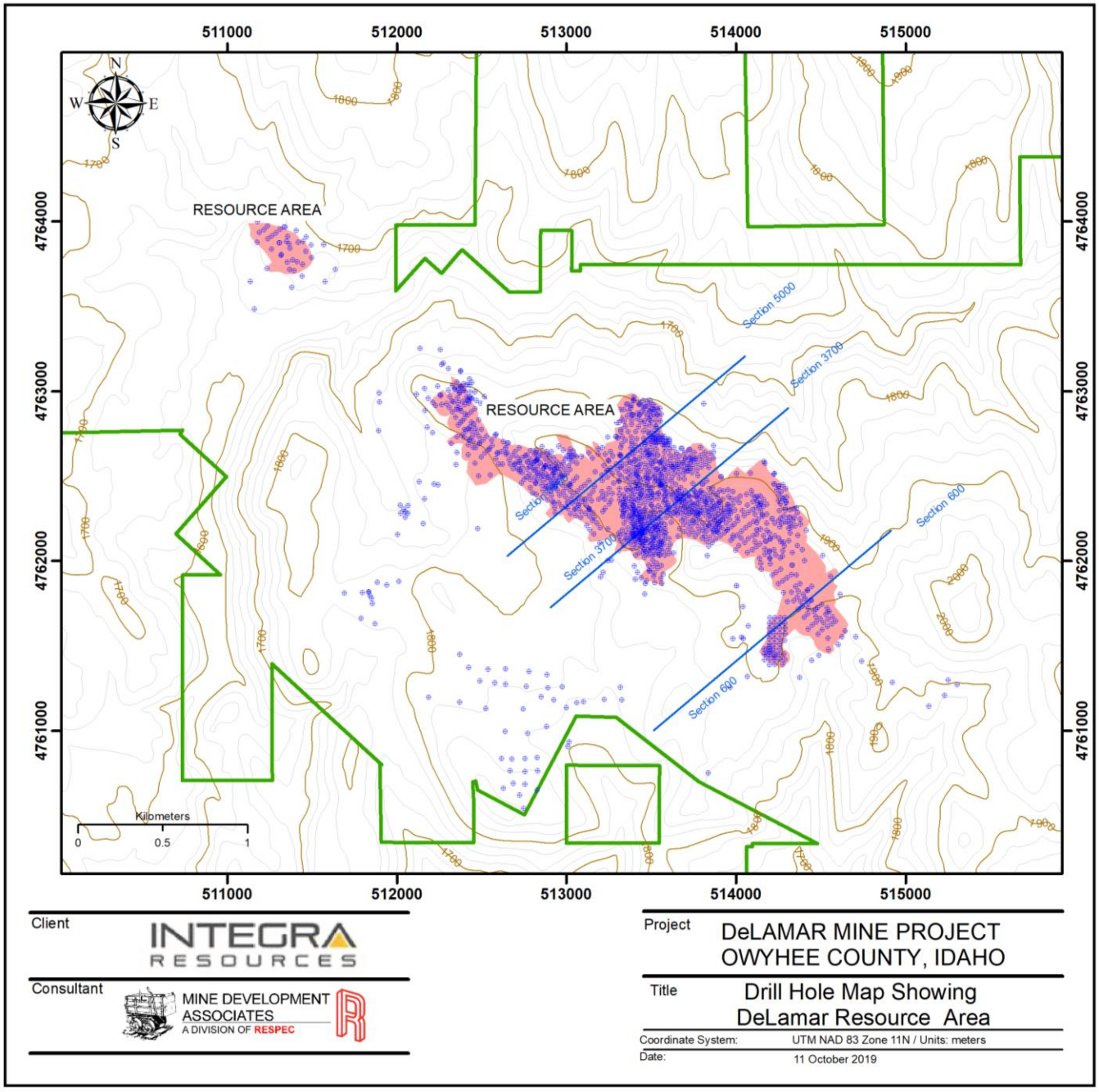
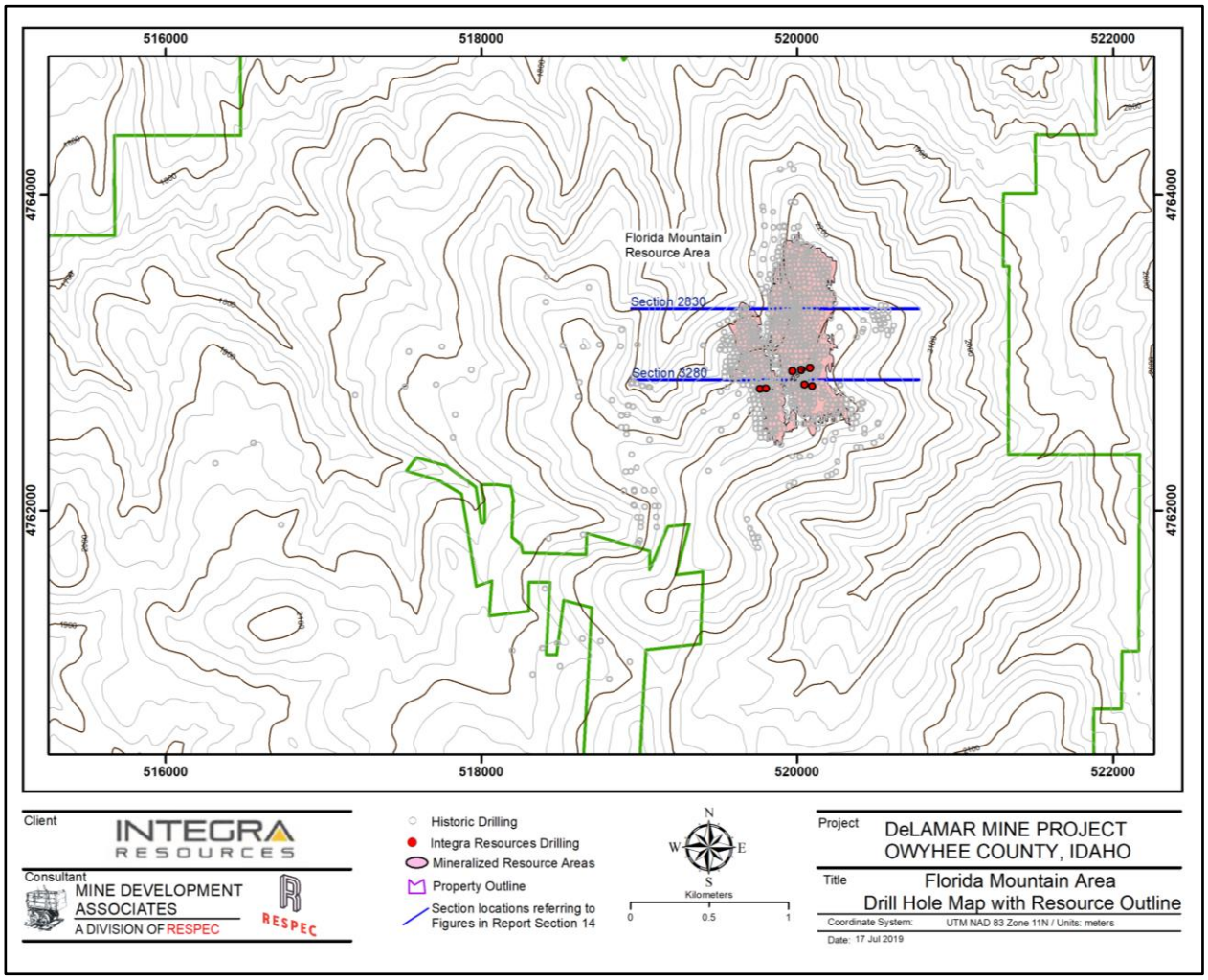


Figure 10.2 Map of Florida Mountain Area Drill Holes



10.2.3 Sidney Mining 1972

Sidney Mining drilled eight core holes in the Sommercamp and North DeLamar zones in 1972. MDA is unaware of the drilling contractor, type of rig, and core diameter(s) used for this drilling.

10.2.4 Earth Resources ~1970 – 1983

Between as early as possibly 1970 and the end of 1983, Earth Resources drilled 465 holes. Five of these were core holes drilled in the DeLamar area in 1975 with Longyear 38 and Longyear 44 core drills operated by Longyear Drilling. Five core holes were also drilled in 1975 in the Glen Silver area by the same contractor using a Longyear 44 rig. The core diameter was HQ for all 10 core holes.

A total of 384 conventional rotary holes, for 659,701 meters, were drilled during this period in the DeLamar, Glen Silver, Sommercamp – Regan, Town Road – Henrietta, Milestone, Ohio, Millsite, and

Sullivan Gulch areas (Figure 6.2). All of these holes were vertical. Contractors at various times included: Justice Drilling using a Mayhew 1000 rig; and Eklund Drilling using G-15, Mayhew 1500, Mayhew 2000, and Gardner-Denver 1500 rigs. Harris Drilling used a Failing drill for 21 holes in 1973. Eklund also used an Ingersoll-Rand TH60 drill in 1979 and 1980, and apparently one of the holes drilled by this rig was a 183-meter vertical RC hole.

Earth Resources drilled an additional 70 vertical holes of unknown type during this period, for a total of 5,202 meters.

10.2.5 NERCO 1985 - 1992

Available records show NERCO drilled 691 holes during 1985 through 1992. These include 351 RC holes for a total of 37,093 meters, seven conventional rotary holes drilled in 1986 for a total of 640 meters, 36 core holes for 1,902 meters, and 28,720 meters of drilling for which the drilling method is not known. 532 of the holes were drilled vertically.

During this period, drilling took place at various times at North DeLamar, Glen Silver, Sommercamp – Regan, Sullivan Gulch, Ohio, Town Road, the tailings area, and an area known as "Heap Leach". The Sullivan Gulch holes were drilled in 1985 or later using RC methods. Twelve vertical RC holes were drilled at the Ohio area, but the rig type and contractor are not available. Six core holes were drilled in the Glen Silver area in 1986 with a Longyear 44 drill. After some point in 1987, all of NERCO's drilling was done with RC methods. Tonto Drilling used an Ingersoll-Rand TH60 RC drill for some of the drilling in 1987 and 1989. An in-house Canterra RC drill was also used in 1989. Ponderosa Drilling was the contractor for 30 core holes drilled in the Heap Leach area in 1990, but the type of drill and core diameter is not known to MDA. The NERCO Cantera RC drill was also used for 19 holes drilled in the Ohio area in 1991, and 19 RC holes drilled in the Ohio and Town Road areas in 1992.

10.2.6 Kinross 1993 - 1998

Kinross drilled 239 holes in the DeLamar area, and only six of these holes were drilled vertically. Kinross drilled 55 RC holes (4,491 meters) in 1993 in the North DeLamar, Glen Silver, and Sommercamp – Regan areas. The drilling contractor was Stratagrout and a Discovery drill was used. In 1994 and 1995, Kinross drilled 181 RC holes (16,624 meters) located in the North DeLamar, Glen Silver, Ohio, and Sommercamp – Regan areas. AK Drilling was the contractor for 19 of these holes, and Drilling Services was the contractor for at least six of the holes. Available records indicate only one 158-meter inclined RC hole was drilled in 1996, and two additional inclined RC holes, for a total of 91 meters, are bracketed to have been drilled between 1995 to 1998.

10.3 Historical Drilling – Florida Mountain Area

10.3.1 Earth Resources 1972 – 1976

During 1972, 1975 and 1976, Earth Resources drilled a total of 3,405 meters in 45 vertical rotary holes in the Florida Mountain area. This drilling was done by Eklund Drilling of Elko, Nevada, using 13.34-centimeter diameter hammer bits. A Gardner-Denver 15 rotary rig was used for the 1972 holes and a

Mayhew 1500 drill was used for the 1975 – 1976 drilling. Samples were collected over 3.048-meter intervals, but MDA is unaware of any other specific drilling and sampling procedures and methods.

10.3.2 ASARCO 1977

ASARCO drilled four vertical rotary holes in 1977 between DeLamar and Florida Mountain in an area that is not presently part of the land controlled by Integra. These holes total 579 meters drilled, and are part of the project database, but were not used to estimate the current mineral resources. Samples were assayed over 3.048-meter intervals, but MDA is unaware of the drilling contractor, type of drill used, or the drilling and sampling procedures and methods.

10.3.3 Earth Resources 1980

In 1980, Earth Resources drilled nine vertical rotary holes at Florida Mountain for a total of 651 meters. Eklund Drilling and D. Allen Drilling were the contractors. A Midway and Ingersoll Rand TH100 drill were used, respectively, with 13.34-centimeter diameter hammer bits. Samples were collected over 1.524-meter intervals, but MDA is unaware of any other specific drilling and sampling procedures and methods.

10.3.4 NERCO 1985 – 1990

NERCO drilled 898 exploration holes at and near Florida Mountain from 1986 through 1990, by far the largest amount of drilling by a single historical operator (Table 10.2). Thirty-six of the holes, for a total of 4,488 meters, were inclined HQ-diameter (63 millimeter) core holes, with the remainder drilled by RC methods (11,729 meters). Twenty-eight of these RC holes were drilled vertically. Incomplete records show that 5 water wells, for a total of 475 meters, were also drilled in 1988. At least one, and possibly all, of the water wells were drilled with a CP650 drill operated by "Allberry". MDA is not aware of the drilling contractor or type of rig that was used to drill the core holes. In 1986, a total of 7,393 meters were drilled in 50 RC holes by Becker Drilling with a Drill Systems rig, but no other information is available on the specific methods and procedures used. MDA is not aware of the drilling contractors, rig types, and drilling methods and procedures used for NERCO's RC drilling in 1987, 1988, 1989 and 1990.

10.3.5 Kinross 1995 – 1997

During 1995 through 1997, Kinross drilled a total of 9,901 meters in 99 RC holes in the Florida Mountain area. All but three of the 99 holes were inclined. Available records suggest that Drilling Services Company ("DSC") of Chandler, Arizona, was the contractor for the three holes drilled in 1995, and that a TH100 drill was used. Dateline Drilling of Missoula, Montana, was the contractor for the 1996 drilling, which totaled 4,907 meters in 49 holes. In 1997, a total of 4,658 meters were drilled in 47 RC holes at Florida Mountain by AK Drilling of Ramsay, Montana, with a Foremost Prospector rig. For the Kinross drilling, samples were collected over 1.524-meter intervals, but MDA is unaware of any other specific drilling and sampling procedures and methods.

10.4 Integra Drilling 2018 - 2019

Integra's drilling is summarized in Table 10.3.

Table 10.3 Integra Drilling Summary

Area/Target	Year	RC Holes	RC Meters	Core Holes	Core Meters	PC Core Holes	PC Core Meters	Total Holes	Total Meters
DeLamar									
DeLamar	2018	1	430	2	609	1	246	4	1,284
DeLamar North	2018			2	311	7	1,437	9	1,748
Glen Silver	2018	2	634	4	1,028			6	1,662
Henrietta	2018	5	1,228					5	1,228
Milestone	2018	6	1,218					6	1,218
Ohio	2018	5	1,689					5	1,689
Sommercamp	2018	1	384	2	641	3	700	6	1,725
Sullivan Gulch	2018	12	4,622	6	2,309			18	6,931
Sullivan Knob	2018	4	1,326					4	1,326
Town Road	2018	2	652					2	652
TruckShop	2018	2	867					2	867
Glen Silver	2019			1	198			1	198
Sullivan Gulch	2019	11	4,947					11	4,947
TruckShop	2019	4	1,880	9	3,285			13	5,165
DeLamar Total	2018-2019	55	19,877	26	8,380	11	2,383	92	30,640
Florida Mountain	2018			10	2,932			10	2,932
All Integra Drilling	***2018-2019***	***55***	***19,877***	***36***	***11,313***	***11***	***2,383***	***102***	***33,573***

10.4.1 DeLamar Area Drilling 2018 - 2019

A total of 33,573 meters were drilled in 102 holes in various parts of the DeLamar area in 2018 and through April 2019. Approximately 60% of the holes and 65% of the meters were drilled with RC methods. The balance of the DeLamar area holes were drilled with diamond core, or with an initial RC "pre-collar" followed by a core tail. Only one of the 2018 and 2019 DeLamar area holes was vertical, with the others inclined at angles of -45° to -85°.

The RC drilling in 2018 and 2019 was conducted by Boart Longyear of Elko, Nevada using an MPD 1500 track-mounted drill. Bit diameters varied from 12.065 centimeters to 15.558 centimeters (4.75-6.125 inches). RC drilling was conducted wet; samples were passed through a rotating vane-type splitter to obtain samples generally in the range of 4.54 kilograms to 9.07 kilograms when dry. The RC samples were transported from the drill pads to the on-site logging and storage facility each day.

In 2018, the core holes were drilled by Major Drilling of Salt Lake City, Utah using LF90 track-mounted drill. HQ- and lesser PQ-size core was recovered with wireline methods that involved triple-tube coring.

The 2019 core drilling at DeLamar was conducted by Boart Longyear of West Valley City, Utah using a track mounted LF90 core rig. PQ- and lesser HQ-size core was recovered with wireline methods and triple-tube coring.

The 2018 and 2019 drill core was placed in plastic core boxes by the drilling contractor and transported from the drill sites to Integra's secure sample logging and storage area at the historical DeLamar mine site on a daily basis.

10.4.2 Florida Mountain Area Drilling 2018

In the Florida Mountain area, a total of 2,932 meters were drilled in 10 core holes (Table 10.3). These holes were inclined at angles of -50° to -75°. The core holes were drilled by Major Drilling of Salt Lake City, Utah using LF90 track-mounted drill. HQ- and lesser PQ-size core was recovered with wireline methods that involved triple-tube coring. The drill core was placed in plastic core boxes by the drilling contractor and transported from the drill sites to Integra's sample logging and storage area at the DeLamar mine.

10.5 Drill-Hole Collar Surveys

Nearly all historical drill-hole collar locations were surveyed in local mine-grid coordinates by one or more dedicated mine surveyors. It is Mr. Gustin's and Mr. Weiss' understanding that the mine-grid coordinate system was established in the 1970s by Earth Resources' surveyors. Mine-grid coordinate 100,000 East and 50,000 North is located at the surveyed Section corner between Sections 32 and 33 of Township 4 South, and Sections 4 and 5 of Township 5 South, on the hillside north of the De Lamar town site. The exact surveying procedures and type of equipment used to survey hole locations are not known to Mr. Gustin and Mr. Weiss. Surveyed hole coordinates were hand recorded in multiple copies of collar coordinate logbooks. The logbooks show that coordinates for 44 holes were "*taken from maps*". These are from several different areas of drilling and are mainly the older holes drilled in those areas.

The x and y collar locations of Integra's 2018 and 2019 drill holes were surveyed by Integra geologists using a Bad Elf GPS. The measured coordinates were then processed using the Natural Resources Canada website. Based on check surveys of post-processed Bad Elf GPS coordinates, Integra found the accuracy at the project to be less than one meter, usually considerably less. Elevations were assigned to each of the post-processed GPS x and y coordinates using the LiDAR data (see Section 9.1).

10.6 Down-Hole Surveys

None of the historical RC and conventional rotary holes in the DeLamar area are known to have been surveyed for down-hole deviations, while only 33 RC holes drilled in the Florida Mountain area have down-hole survey information in the database. Conventional rotary and RC drill holes can deviate significantly, in both dip and azimuth, with increasing deviations as depths increase, particularly in the case of inclined holes. It is therefore likely that deviations occurred in the historical drill holes at the DeLamar project.

Integra used a REFLEX EZ-GYRO EG0270 down-hole survey tool to measure down-hole deviation in the 2018 and 2019 drill holes. The instrument was operated by Integra personnel to survey the RC holes, and by Major Drilling and Boart Longyear drillers to survey the core holes. Azimuth, dip, and temperature were measured at 15.24-meter (50-foot) intervals. A few of the 2018 drill holes were also surveyed down hole with an optical and acoustic tele-viewer system, although this information was not used in the project database.

10.7 Sample Quality and Down-Hole Contamination

Down-hole contamination is always a concern with holes drilled by rotary (RC or conventional) methods. Contamination occurs when material originating from the walls of the drill hole above the bottom of the hole is incorporated with the sample being extracted at the bit face at the bottom of the hole. The potential for down-hole contamination increases substantially if significant water is present during drilling, whether the water is from in-the-ground sources or injected by the drillers. Conventional rotary holes, in which the sample is returned to the surface along the space between the drill rods and the walls of the drilled hole, are particularly susceptible to down-hole contamination.

Some of the drill-hole logs reviewed by Mr. Gustin and Mr. Weiss were found to have notations as to the presence of water during drilling, as well as occasional comments concerning drilling difficulties and sample sizes. Integra has comprehensively compiled sample quality information from the historical drill logs, and this information, which includes logged notes on intersected groundwater and/or drill-injected fluids, was used by MDA in the modeling of project resources. For example, intervals for which down-hole contamination was noted or suspected by historical operators were evaluated in the context of surrounding holes, and when such intervals were deemed by MDA to have suspicious results, they were excluded from use in the resource estimation. Intervals noted as having poor recovery were treated similarly. Beyond these historical notations of possible contamination, MDA noted other historical drill intervals that likely experienced down-hole contamination, and these intervals were excluded as well.

Down-hole contamination is not a significant issue with the historical drilling at the DeLamar project due to the relatively shallow depths of these holes (median down-hole depths of 91 meters for the mostly vertical holes in the DeLamar area and 123 meters for the predominantly angled holes at Florida Mountain). A few of the deeper Integra RC holes, which penetrated to depths significantly below the water table, do have strong evidence of down-hole contamination, and these intervals were removed from use in estimation of the resources.

10.8 Summary Statement

There is a complete lack of down-hole deviation survey data for the historical holes in the DeLamar area database, and the Florida Mountain area database includes deviation data for 33 RC and four core holes. While the paucity of such data is not unusual for drilling done prior to the 1990s, the lack of deviation data contributes a level of uncertainty as to the exact locations of drill samples at depth. However, in the DeLamar area these uncertainties are mitigated to a significant extent by the vertical orientation of three-quarters of the drill holes, the generally shallow down-hole depths, and the likely open-pit nature of any potential future mining operation that is based in part on data derived from the historical holes. Such uncertainties, while still minor, are more pronounced in the Florida Mountain area, where about 81% of the holes were inclined, and the holes were generally slightly deeper than those in the DeLamar area.

Down-hole lengths of gold and silver intercepts derived from vertical holes, which were almost exclusively historical holes, can significantly exaggerate true mineralized thicknesses in cases where steeply dipping holes intersect steeply dipping mineralization, for example in portions of the Sommercamp area. This effect is entirely mitigated by the modeling techniques employed in the estimation of the current resources, however, which constrain all intercepts to lie within explicitly interpreted domains that appropriately respect the known and inferred geologic controls and mineralized thicknesses as evident from the drill data.

The overwhelming majority of sample intervals in the DeLamar and Florida Mountain databases have a down-hole length of 1.52 meters (five feet). This sample length is considered appropriate for the near-surface style of mineralization that characterizes the current mineral resources at both the DeLamar and Florida Mountain areas.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

This section summarizes all information known to Mr. Gustin and Mr. Weiss relating to sample preparation, analysis, and security, as well as quality assurance/quality control procedures and results, that pertain to the DeLamar project. The information has either been compiled by Mr. Gustin and Mr. Weiss from historical records as cited or provided by Ms. Richardson, a longtime employee at the mine. Ms. Richardson's contributions to this section are derived from personal correspondences with Mr. Gustin and Mr. Weiss, an internal mine memorandum by Richardson (1985), and a recent informal summary document compiled at the request of MDA.

11.1 Historical Sample Preparation and Security

Mr. Gustin and Mr. Weiss are not aware of sample-preparation procedures or sample-security protocols employed prior to the start-up of open-pit mining operations in 1977, although further detailed reviews of historical documentation may yield such information in the future.

Elkin (1993) stated that sample preparation procedures at the mine laboratory had remained relatively constant up to the date of his ore-reserve report. Drill cuttings were split at the drill site to obtain samples weighing approximately 4.5 kilograms. When received at the mine laboratory, the samples were dried and crushed to -10 mesh. Splits of 150 milliliter volumes were then pulverized to pulps with 90% passing 100 mesh. At the date of Elkin's report, one-assay-ton (30-gram) aliquots were taken from these pulps for assaying.

Mr. Gustin and Mr. Weiss are unaware of any specific sample-security protocols undertaken during the various historical drilling programs at the DeLamar project. However, approximately 75% of the drill data in the DeLamar area database and 98% of the holes in the Florida Mountain area are derived from drilling undertaken after the open-pit mining operations had initiated. It is very likely that the drilling and sampling completed during the mining operations was undertaken in areas of controlled access.

11.2 Integra Sample Handling and Security

Integra's RC and core samples were transported by the drilling contractor or Integra personnel from the drill sites to Integra's logging and core cutting facility at the DeLamar mine on a daily basis. The RC samples were allowed to dry for a few days at the drill sites prior to delivery to the secured logging and core-cutting facility.

The 2018 and 2019 core sample intervals were sawn lengthwise mainly into halves after logging and photography by Integra geologists and technicians in the logging and sample storage area. In some cases, the core was sawed into quarters. Sample intervals of either ½ or ¼ core were placed in numbered sample bags and the remainder of the core was returned to the core box and stored in a secure area on site. Core sample bags were closed and placed in a secure holding area awaiting dispatch to the analytical laboratory.

All of Integra's rock, soil and drilling samples were prepared and analyzed at American Assay Laboratories ("AAL") in Sparks, Nevada. AAL is an independent commercial laboratory accredited through February 1, 2020 to the ISO/IEC Standard 17025:2005 for testing and calibration laboratories.

The drilling samples were transported from the DeLamar mine logging and sample storage area to AAL by Integra's third-party trucking contractor.

The soil samples were screened to -80 mesh for multi-element analysis at AAL. MDA has no other information on the methods and procedures used for the preparation of Integra's soil and rock samples.

11.3 Historical Sample Analysis – Prior to Commercial Open-Pit Mining Operations

Prior to the opening of the mine in April 1977, all gold and silver analyses of drill-hole samples consisted of fire assays completed by commercial laboratories, primarily Union Assay Office of Salt Lake City, Utah ("Union Assay"). This includes the core holes drilled by Continental in 1966 and Sidney Mining in 1972, as well as pre-mining Earth Resources drilling. Assay certificates from other commercial laboratories reviewed by Mr. Gustin and Mr. Weiss from this time period include those from Rocky Mountain Geochemical Corp. of Salt Lake City, Utah ("RMGC") and Western Laboratories in Helena, Montana. Several holes were also found to have had samples analyzed by Earth Resources Naciamento Copper Mine Laboratory ("Earth Resources Lab"), which apparently was an internal laboratory in Cuba, New Mexico operated by Earth Resources. Mr. Gustin and Mr. Weiss know of no other details of the sample analyses performed prior to the beginning of mining operations in April 1977.

11.4 Historical Sample Analysis – During Commercial Open-Pit Mining Operations

Upon initiating mining operations in April 1977, all ore-control (blast-hole) samples and most samples from exploration and development drilling were assayed at the DeLamar mine laboratory. Until approximately 1988, these in-house analyses were completed by MIBK atomic absorption ("AA") methods (Porterfield and Moss, 1988). Gold was solubilized from 20 grams of material using an unspecified method and then extracted from the solution using methyl isobutyl ketone (MIBK), with the gold concentration determined by AA. Approximately 60% of the historical drill holes in the DeLamar area database and 28% of those in the Florida Mountain area holes were drilled prior to 1988.

From approximately 1988 through to the end of the open-pit mining operations, all analyses by the mine laboratory were completed using standard fire-assay methods. Records reviewed by Mr. Gustin and Mr. Weiss reveal that some samples during this period were analyzed by Chemex Laboratories, Inc. of Reno, Nevada; RMGC; Union Assay; Legend Inc. of Reno, Nevada; Western Laboratories; and Earth Resources Lab. Union Assay and RMGC were most commonly used. According to Ms. Richardson, all gold and silver analyses were completed by fire assay with a gravimetric finish. The mine lab used silver in quarts to measure gold and silver gravimetrically.

Repeat fire assays by the mine laboratory of samples prior to 1988 that were originally analyzed by AA at the mine laboratory showed that the silver AA results were consistently lower than the fire assays, sometimes significantly lower; although fire-assay checks of the AA gold results were stated to have compared well. The mine laboratory staff believed that the understatement of the silver AA values was due to a relatively coarse grind in the sample preparation, which ultimately resulted in incomplete digestion of silver-bearing minerals prior to the AA analyses. Sometime in 1980, the mine instituted a much more systematic check-assay program, whereby sets of silver-mineralized samples from each mine area, as defined by mine AA analyses, as well as from certain ranges of mine benches within a

mine area, were selected for checking by fire assay. The AA and fire-assay analyses were then compared by area, and a linear factor was determined that was used to mathematically increase the AA values for each area or set of benches analyzed. Factored silver values of blast-hole samples were used by the mining operation to determine waste from ore. Silver AA adjustment factors were also determined for each developmental drilling area until 1985, when it appears that factoring of the silver AA values ended.

The systematic fire-assay check program was continuously monitored, with changes to the silver adjustment factors occurring frequently. Documents reviewed by Mr. Gustin and Mr. Weiss indicate that the factor was subject to modification as frequently as once monthly for each active mining or developmental drilling area.

Ms. Richardson stated that the factoring of the blast-hole silver AA analyses worked well, as evidenced by the reported close agreement between mined grades determined by blast-hole data and head grades determined at the mill.

Because the Florida Mountain area was mined from 1994 to 1998, all gold and silver of blast holes, and most of the drill holes as well, were analyzed by fire assaying methods. According to Ms. Richardson, a silver in quart was added prior to fire assaying due to the generally low silver concentrations at Florida Mountain relative to the DeLamar area.

In 1997, Kinross also shipped 1,691 Florida Mountain RC drill intervals to Legend Inc. in Reno, Nevada, for sample preparation and assays of gold and silver. The samples were crushed to nominal 10 mesh, then split to obtain a 200-gram sub-sample that was pulverized to nominal 200 mesh pulp. Gold and silver were determined on 30-gram aliquots using fire-assay fusion with a gravimetric finish.

No further details of the sample analyses completed during open-pit mining operations are known to Mr. Gustin and Mr. Weiss.

11.5 Integra Sample Analysis

The same principal analytical methods were used at AAL for both soil and surface-rock samples collected by Integra. Gold was determined by fire-assay fusion of 60-gram aliquots with an inductively coupled plasma optical-emission spectrometry ("ICP") finish. Silver and 44 major, minor and trace elements were determined by ICP and mass spectrometry ("ICP-MS") following a 5-acid digestion of 0.5-gram aliquots. Rock samples that assayed greater than 10 g Au/t were re-analyzed by fire-assay fusion of 30-gram aliquots with a gravimetric finish. Samples with greater than 100 g Ag/t were also re-analyzed fire-assay fusion of 30-gram aliquots with a gravimetric finish. Some rock samples were analyzed for gold using a metallic-screen fire assay procedure.

RC samples from the 2018 and 2019 drilling were dried upon arrival at AAL's Reno facility. The dry samples were crushed to a size of -6 mesh and then roll-crushed to -10 mesh. One-kilogram splits of the -10-mesh materials were pulverized to 95% passing -150 mesh. Sixty-gram aliquots of the one-kilogram pulps were analyzed at AAL for gold mainly by fire-assay fusion with an ICP finish. Silver and 44 major, minor, and trace elements were determined by ICP and ICP-MS following a 5-acid digestion of 0.5-gram aliquots. Samples that assayed greater than 10 g Au/t were re-analyzed by fire-

assay fusion of 30-gram aliquots with a gravimetric finish. Samples with greater than 100 g Ag/t were also re-analyzed fire-assay fusion of 30-gram aliquots with a gravimetric finish. Selected RC samples were analyzed for gold using a metallic-screen fire assay procedure.

Integra's 2018 and 2019 core samples were prepared and assayed at AAL for gold, silver, and multi-elements using the identical methods used for the RC samples.

11.6 Quality Assurance / Quality Control Programs

Quality Assurance / Quality Control ("QA/QC") programs undertaken as part of the various exploration and development drilling programs of historical operators and Integra are described in this subsection. The results of these programs are discussed in Section 12.2.1 and 12.2.2, respectively.

11.6.1 Historical Operators

Approximately 25% of the historical exploration and development holes in the DeLamar area and 4% of the holes in the Florida Mountain area were drilled prior to the initiation of open-pit mining and the use of the mine-site analytical laboratory. In this time prior to the mining operations, quality assurance/quality control ("QA/QC) procedures were employed to monitor Union Assay's analytical results, but these QA/QC data, which exist in paper form, need to be compiled digitally and then evaluated independently. The results of the mine laboratory were monitored by resubmitting samples to the mine laboratory for check assaying, but documentation of these check analyses is incomplete.

According to the 1974 historical feasibility study (Earth Resources Company, 1974), the Union Assay results obtained prior to the initiation of open-pit mining were checked by sending composites of Union Assay pulps, splits of drill core, and Union Assay coarse rejects to the following laboratories: Southwestern Assayers and Chemists in Tucson, Arizona; Skyline Laboratories in Denver, Colorado; Western Laboratories in Helena, Montana; Hazen Research in Golden, Colorado; and the Earth Resources Lab in Cuba, New Mexico. The various check samples were analyzed by either fire assay or atomic-absorption methods.

The Elkin (1993) report, confirmed by Ms. Richardson, indicated that repeat (check) assays were routinely run at the mine laboratory. Elkin reported that all samples with silver values in excess of 10 ounces per ton (343 g/t) or gold values greater than 0.1 opt (3.43 g/t) were resubmitted to the mine laboratory for check assaying. The assay pulp and a separate split from every fourteenth sample were also resubmitted to the mine laboratory on a routine basis. Elkin also stated that duplicate samples were not being sent to outside laboratories at the time of his report. Mr. Gustin and Mr. Weiss have not found detailed documentation of these check analyses, and therefore could not independently evaluate the results.

The mine lab completed duplicate MIBK analyses and/or fire assays as a check on the MIBK results. Samples with gold concentrations greater than 0.02 ounces per ton (0.7 g Au/t) and those within "geologically interesting zones" were fire assayed by outside commercial laboratories using 60-gram charges. The mine lab performed checks of the outside lab results, using fire assaying techniques on 30-gram charges. Porterfield and Moss reported that these checks verified the results of the commercial labs.

During 1997, Kinross shipped a total of 1,134 pulps of exploration RC drill samples from Florida Mountain to Legend Inc., in Reno, Nevada, for check assaying of gold and silver. The samples had apparently been crushed, split, and pulverized in the DeLamar mine laboratory. At Legend, the pulps were analyzed by fire-assay fusion with gravimetric finish using 30-gram aliquots. The results of this program have not been found in the historical documents to date.

11.6.2 Integra

Coarse blank material, certified reference materials ("CRMs"), and RC field duplicates were inserted into the drill-sample streams as part of Integra's quality assurance/ quality control procedures. The coarse blank material consisted of basalt that was inserted approximately every 10th sample. Commercial CRMs were inserted as pulps at a frequency of approximately every 10th sample.

Check assays of original-sample pulps, which are also part of Integra's QA/QC program, have not yet been completed.

11.7 Summary Statement

None of the analytical laboratories used during historical exploration and mining operations mentioned in this section were certified, as the formal certification process used today had not yet been implemented. Mr. Gustin and Mr. Weiss are not familiar with Western Laboratories or the Earth Resources Company internal laboratory, and the laboratories of Hazen Research and Southwestern Assayers and Chemists were not commonly used for routine assaying by the mining industry. However, historical documents reviewed by Mr. Gustin and Mr. Weiss indicate that Union Assay and, to a lesser extent, RMGC were the primary commercial laboratories used by all operators prior to Kinross, and these were independent commercial laboratories that were widely recognized and used by the mining industry at that time.

Documentation of the methods and procedures used for historical sample preparation, analyses, and sample security, as well as for quality assurance/quality control procedures and results, is incomplete and in many cases not available. It is important to note, however, that the historical sample data were used to develop and operate a successful commercial mining operation that produced more than 400,000 ounces of gold and 26 million ounces of silver. Mr. Gustin and Mr. Weiss are therefore satisfied that the historical analytical data are adequate to support the current resources, interpretations, conclusions, and recommendations summarized in this report.

Integra's sample preparation and analyses were performed at a well-known certified laboratory, and the sample security and assurance/quality control procedures all met industry norms.

12.0 DATA VERIFICATION

12.1 Drill-Hole Data Verification

The current drill-hole databases, which support the resource estimations of the DeLamar and Florida Mountain areas, were created by MDA using the original, DeLamar mine digital database files obtained from the current mine site for each of these two areas of the project. This original mine-site drill-hole information was then supplemented with Integra's drilling data and results through May 1, 2019. The historical information was subjected to various verification measures, the primary one consisting of auditing of the digital MDA databases by comparing the drill-hole collar coordinates, hole orientations, and analytical information to the original historical paper records in the possession of Integra. Integra's drilling data in the MDA databases was audited by comparing the digital drill-hole collar coordinates, hole orientations, and analytical information to electronic files provided to MDA by Integra, and to laboratory reports of analyses.

The DeLamar area database is comprised of information derived from 1,550 historical holes and 83 Integra holes. A total of 235 of the historical holes were randomly chosen for auditing, and the data for all Integra holes were audited to some level. The database for the Florida Mountain area includes data from 1,075 historical drill holes, of which 169 were audited, and 10 Integra holes that were all subjected to some level of auditing. The results of this work, as well as other forms of verification, are summarized in this Section.

12.1.1 Collar and Down-Hole Survey Data

DeLamar Area. Drill-hole collar location information was found in the historical documentation for 157 of the 235 holes selected for auditing. The locations of two holes were found to have substantially different locations in the project database compared to the paper records; it remains unclear as to which of the two sources is more accurate. A third hole had an 18-meter difference in elevation with the paper records, but the database elevation matches the project topography and is therefore deemed to be more accurate. All other location discrepancies were due to the rounding of surveyed locations documented in paper records to the nearest foot (0.305 meters), or the truncation of surveyed decimals in the mine-site database. These discrepancies, which Mr. Gustin considers immaterial, may reflect the perceived accuracy of the original location data.

There were no down-hole deviation data in the original mine-site database files. Ms. Richardson stated that no down-hole surveys were completed on conventional rotary or RC holes, which predominate the historical holes drilled at DeLamar. Six of the audited holes are core holes, but no deviation data were found in the paper records for these holes either. Azimuth and dip records of the hole collars do exist, however, and no discrepancies were found between the historical paper records and the database.

The collar and hole-deviation surveys of 16 of the 73 Integra holes drilled at the DeLamar area were audited by comparing the information provided to MDA by Integra with original electronic files of the surveys; no discrepancies were found.

Florida Mountain Area. Original x-y-z collar location data were found for 74 of the holes chosen for auditing. Three of these were found to have significant x-y discrepancies due to an updated survey

location found in the historical records that was not entered into the original mine-site database. However, the holes as located in both the mine-site database and the updated survey information lie to the south of the modeled resources. No discrepancies were found in the azimuth and dips of the audited holes.

The drill-hole collar locations and down-hole surveys of two of the 10 holes drilled by Integra at Florida Mountain were checked in a similar manner as described above for DeLamar. No discrepancies were found.

12.1.2 Assay Data

Historical Assays. Historical paper records, including copies of original assay certificates, handwritten mine-lab assay sheets, and, to a lesser extent, handwritten assay values included on geologic logs, were used to audit the database assay values from the historical holes. Documentation was found for 154 of the 235 historical holes selected to be audited in the DeLamar area database, and this led to the checking of 9% of all sampled and assayed historical intervals in this database. Discrepancies between the MDA database and paper records that are unrelated to the treatment of lower-than-detection-limit results, or unanalyzed intervals, were found in only nine of the 7,758 sample intervals audited, and less than half of these discrepancies are material. As part of this verification process, analytical data from a total of 195 historical sample intervals were found that were not included in the original database; these data were added to the current project database.

Historical back-up data for the gold and silver values of 141 of the holes selected for auditing from the Florida Mountain area were found, representing 13% of the historical holes in the database and 12% of the historical sample intervals. A sequence error was found in which gold and silver values for one sample interval were repeated in the next sample interval, and the following gold and silver values were shifted down one interval for the next eight samples. The affected intervals are very low grade, except for a single 0.41 g Au/t value. In addition to this sequence error, one apparent transcription error was found, whereby the mine-site database has a value of 1.81 oz Ag/ton (62 g Ag/t) versus a value of 0.813 oz Ag/ton (30 g Ag/t) on the original assay sheet.

Analytical data for 41 historical sample intervals in two holes drilled at Florida mountain were found and added to the current project database as a result of the auditing.

Integra Assays. Integra provided the MDA with a complete assay compilation for all holes drilled in 2018 and 2019 through May 1, 2019. The sample numbers in these files were then linked by MDA to original laboratory digital assay certificates to comprehensively validate the Integra assay tables by comparing all Integra assays to the original laboratory certificates. No discrepancies were found during this checking other than in a few cases where MDA chose to use a certain analytical method when multiple methods were available, and the method chosen by MDA differed from that in the Integra compilation.

12.1.3 Integra Data Verification

In addition to MDA checking of historical data using historical records, Integra independently verified the accuracy of the ‘from’, ‘to’, and assay values of a number sample intervals using the MDA’s audited

database, as described in Section 9.3. The very few discrepancies found by Integra were then corrected in the resource databases.

12.2 Quality Assurance/Quality Control Results

The QA/QC programs undertaken by historical operators and Integra are described in Section 11.6 and Section 11.6.2, respectively. The results of these programs are evaluated below.

12.2.1 Historical QA/QC Results

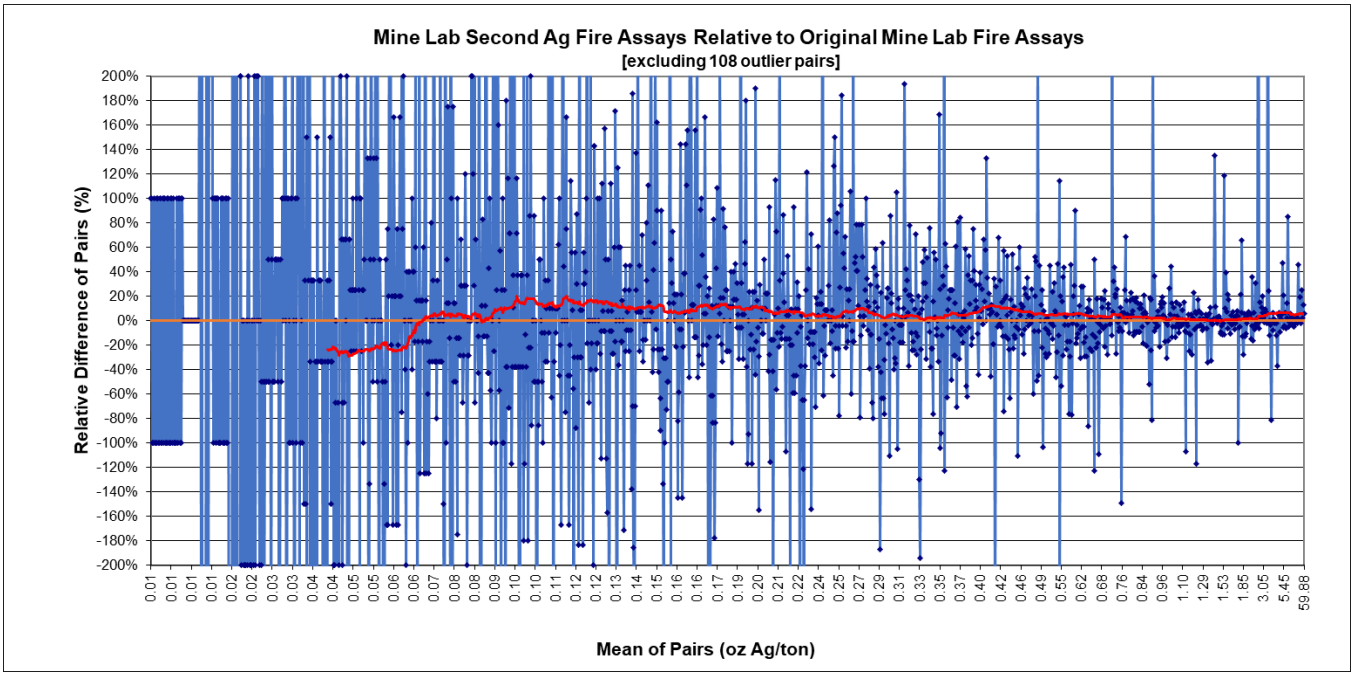
The check-assay results by various third-party labs of the Union Assay analyses that were completed prior to the open-pit mining operation have not been compiled in digital form from the 1974 historical feasibility study (Earth Resources Company, 1974). An evaluation of the check assaying program summarized in the historical feasibility documents concluded that, "*Some variation does exist between the different firms, and since all are generally quite reliable, it is really impossible to determine which one is the best; fortunately, the variations are within reason and appear to fall within a normal and acceptable range of difference.*"

The mine laboratory completed up to three analyses on certain samples, although the nature of the material re-assayed (pulps, coarse rejects, field duplicates) is not known. All available duplicate mine-lab analyses were compiled from the historical databases and evaluated by Mr. Gustin. MDA reviewed the data for 1,762 drill samples for which both primary silver fire assays and second silver fire assays were performed by the mine lab, and both analyses are not below the detection limit. These data are summarized in Figure 12.1, which is a relative-difference graph. The graph shows the percentage difference (plotted on the y-axis) of each duplicate assay relative to its paired primary-sample analysis by the mine lab. The relative difference ("RD") is calculated as follows:

$$100\,x\,\frac{(duplicate\,-\,original)}{lesser\ of\ (duplicate, original)}$$

Positive RD values indicate that the duplicate-sample analysis is greater than the primary-sample assay, while a negative value indicates the duplicate analysis is lower. The x-axis of the graph plots the means of the silver values of the paired data (the mean of the pairs, or "MOP") in a sequential but non-linear fashion. The red line shows the moving average of the RDs of the pairs, which provides a visual guide to trends in the data that can aid in the identification of potential bias. A total of 108 pairs characterized by unrepresentatively high RDs have been excluded from the graph.

Figure 12.1 Repeat Mine Lab Silver Assays Relative to Original Mine Lab Assays



Grades on this graph, and others below showing historical data, are presented in ounces-per-ton, reflecting the original analyses.

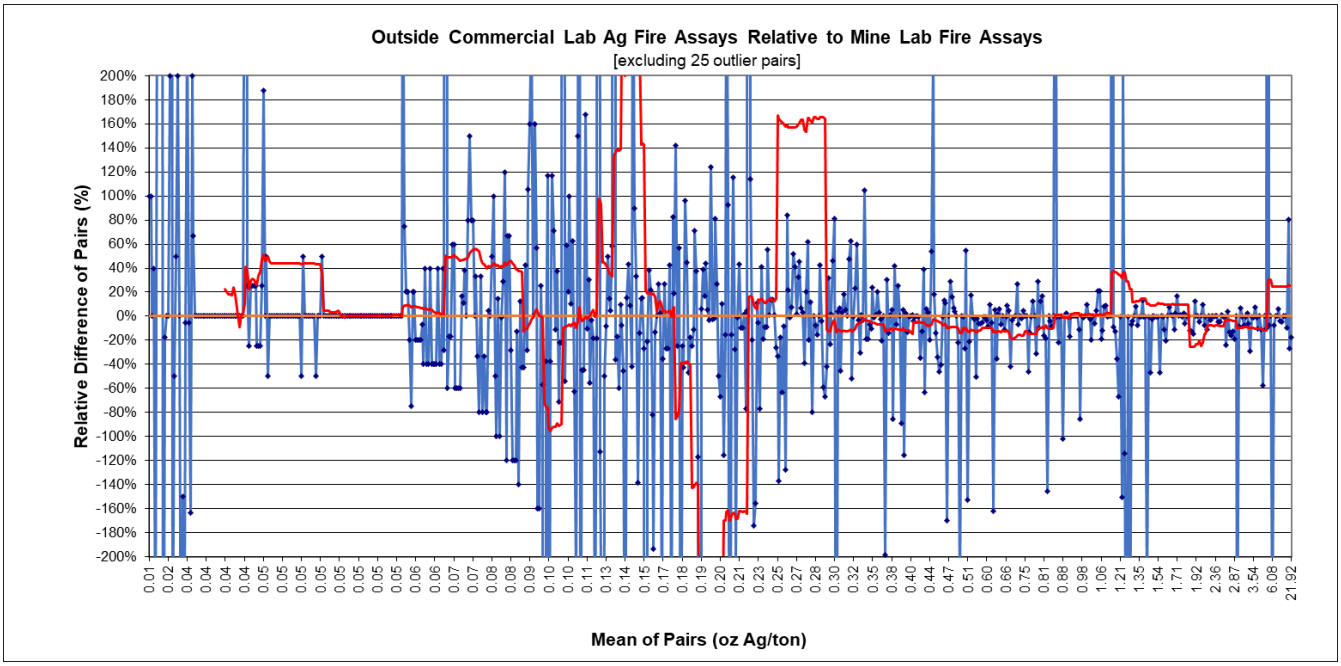
The graph suggests a high bias of low magnitude in the duplicate silver results relative to the original assays over most of the grade range of the data. The mean of duplicate analyses is 0.613 oz Ag/ton (21.0 g Ag/t) is 4% higher than that of the original results (0.588 g Ag/t; 20.2 g Ag/t), and the average RD of the pairs is +2% (the average RD can be an approximate measure of the degree of bias, although one must be aware of the statistical effects of pairs with anomalously high RDs). The mean of the absolute value of the RDs ("AVRD"), a measure of the average variability exhibited by the paired data, is quite high at 73%. At a MOP cutoff of 1.0 oz Ag/ton (34.3 g Ag/t), the mean of the duplicate analyses of the 196 pairs is 5% higher than the original analyses, the average RD is +6%, and the mean AVRD drops to 16%. It should be noted that this high bias in the duplicates relative to the original analyses is present in what is a low-grade dataset.

A similar dataset for 1,837 pairs of gold fire assays, after removal of 15 pairs that exhibit extreme variability, yields identical means (0.013 oz Au/ton; 0.45 g Au/t) for the duplicate and original analyses, an average RD of +1%, and a mean AVRD of 26%. The grades in this dataset are much more representative of the mineralization of interest than the silver duplicate data presented above.

As discussed in Section 11.0, various check analyses of the original mine-lab assays were performed by various commercial, or "outside", laboratories, primarily Union Assay and RMGC. Excluding 25 outlier pairs and all pairs in which the original and check assays were less than the detection limits, a total of 696 pairs of silver fire assays were evaluated. The nature of the material sent to the outside labs for analysis (pulps, coarse rejects, or field duplicates) is not known, nor is the identity of outside lab that

performed the check analyses known, although it is believed that Union Assay completed most of them. These unknowns hinder the analysis. However, the mean of the outside lab duplicates (0.676 oz Ag/ton; 23.2 g Ag/t) is 7% lower than the mean of the original mine lab analysis for the complete dataset, and 8% lower at a cutoff of 1.0 oz Ag/ton (cutoff of 34.4 g Ag/t). The relative difference graph of the data (Figure 12.2) indicates that this discrepancy is largely caused by the prevalence of high-variable pairs having low values for the outside lab relative to the mine lab. Once again, it is also important to note the low-grade nature of the dataset. The moving-average line is of limited use in this case due to the effects of the numerous high-variability pairs.

Figure 12.2 Outside Lab Silver Assays Relative to Original Mine Lab Assays



Only 28 outside lab fire assays for gold were found that were also assayed by the mine lab. The mean of the outside lab analyses is 0.005 oz Au/ton (0.17 g Au/t), while the mine lab assays averaged 0.006 oz Au/ton (0.21 g Au/t).

As discussed in Section 11.4, mine lab AA silver analyses were reported to have been systematically low due to sample digestion issues. Figure 12.3 compares data from 4,378 pairs of mine-lab fire assays and AA analyses. A clear systematic bias is evident, whereby the AA analyses are lower than the paired fire assays. The mine site attributed this to incomplete digestions of silver minerals in the AA analyses, and used the fire-assay data to factor the AA results for use in the mining operations. While the results of the relative difference graph were expected, this was not necessarily the case for the relatively constant magnitude of the low bias. This constancy of the low bias is seen visually in the relative-difference graph, and it is evidenced statistically whereby the mean of the AA analyses is 22% lower than the fire-assay mean for all data and at the several MOP cutoffs inspected. The average RD also is more-or-less constant at all cutoffs at approximately -30%. No AA silver analyses were used in the estimation of the project resources.

Figure 12.3 Mine Lab Silver AA Analyses Relative to Mine Lab Silver Fire Assays

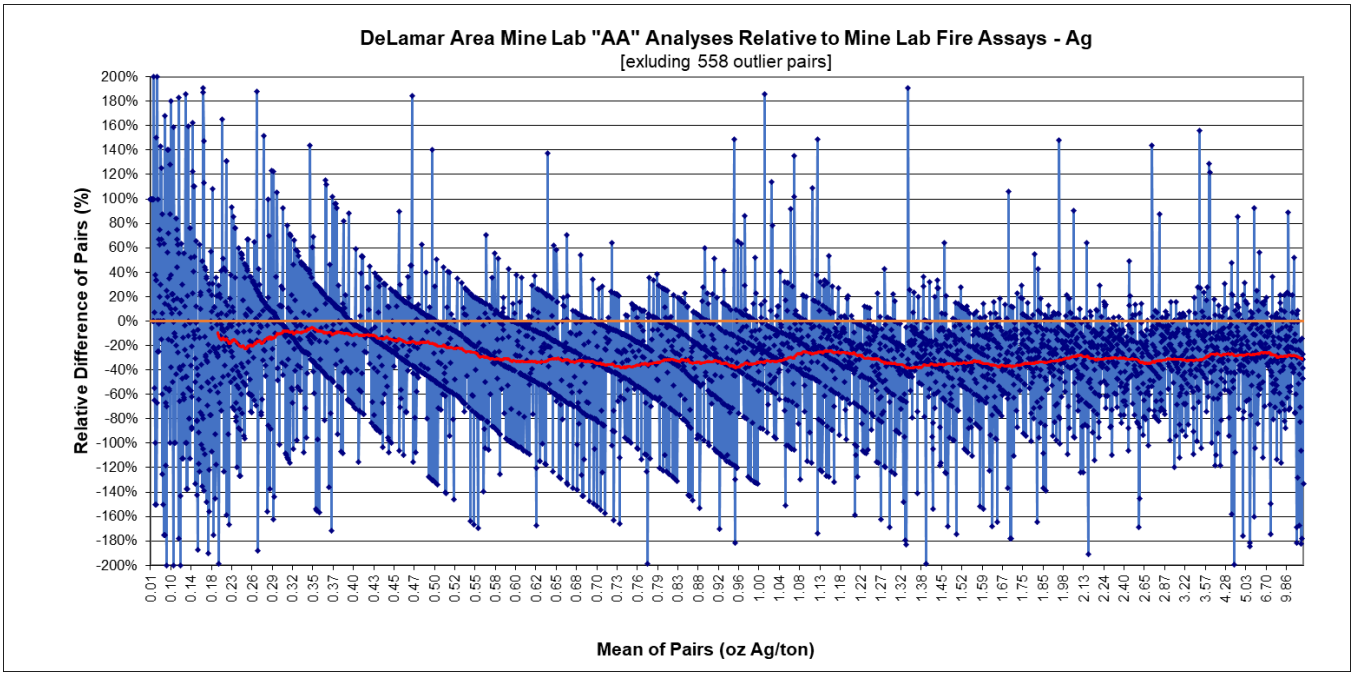
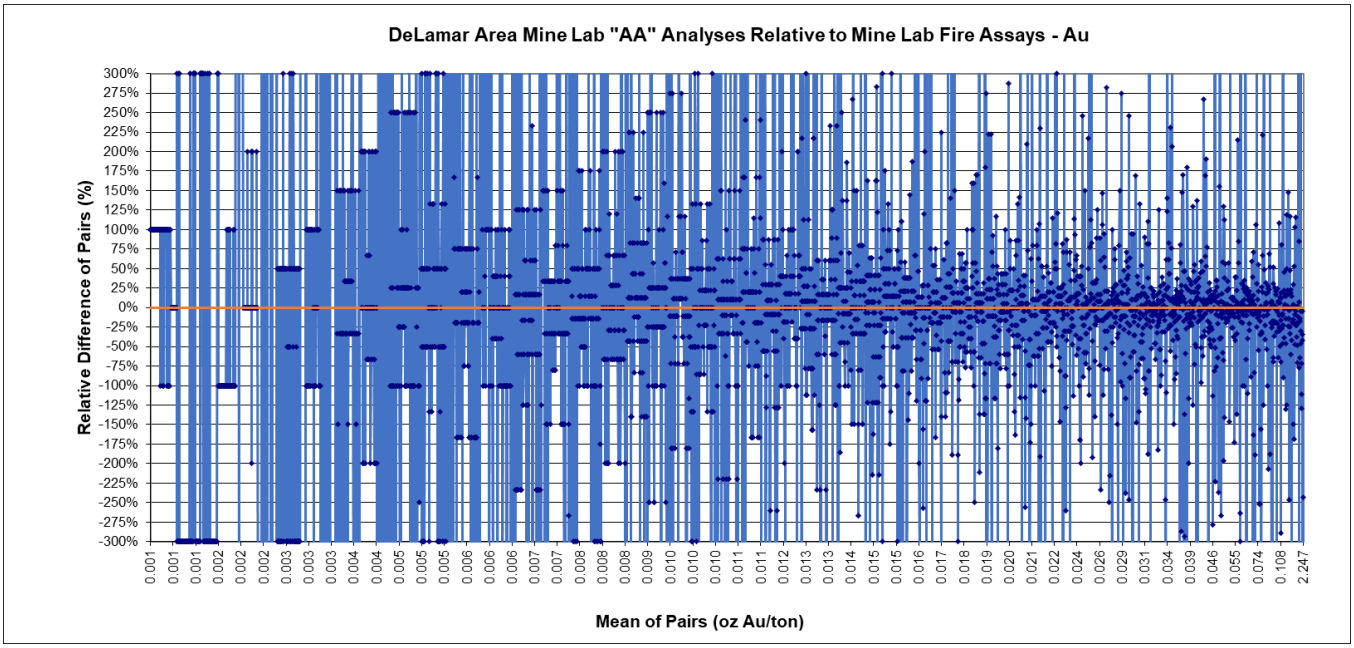


Figure 12.4 compares mine-lab gold AA analyses with mine-lab fire assays of samples from the same intervals.

Figure 12.4 Mine Lab Gold AA Analyses Relative to Mine Lab Gold Fire Assays



All 4,797 pairs are shown, including many pairs with very high variability (the average AVRD is 323%). While the mean of the AA analyses for the entire dataset is 17% lower than that of the fire assays, the means are identical for all MOP less than 0.1 oz Au/ton (3.43 g Au/t). The mean of the AA analyses for the 111 pairs with MOP $\geq$ 0.1 oz Au/ton is 45% lower than the mean of the fire assays. This demonstrates that the difference in the means for the entire dataset is due solely to differences in the highest-grade portion of the data. Accordingly, higher-grade gold values may be understating the actual grades of the samples in cases where mine-site AA analyses are the only available assay, which would therefore lead to their use in the resource estimations.

12.2.2 Integra QA/QC Results

CRMs. Integra purchased commercial CRMs (certified reference materials) for use in their 2018 and ongoing 2019 drilling programs. The CRMs were inserted into the primary sample stream and analyzed with the drill samples. The results were used to evaluate the analytical accuracy and precision of the AAL analyses of Integra's drill samples.

In the case of normally distributed data, 95% of the CRM analyses are expected to lie within the two standard-deviation limits of the certified value, while only 0.3% of the analyses are expected to lie outside of the three standard-deviation limits. Note, however, that most assay datasets from metal deposits are positively skewed. Samples outside of the three standard-deviation limits are typically considered to be failures. As it is statistically unlikely that two consecutive analyses of CRMs would lie between the two and three standard-deviation limits, such samples are also considered to be failures unless further investigations suggest otherwise. All potential failures should trigger investigation, possible laboratory notification of potential problems, and possible reanalysis of all samples included with the failed standard result.

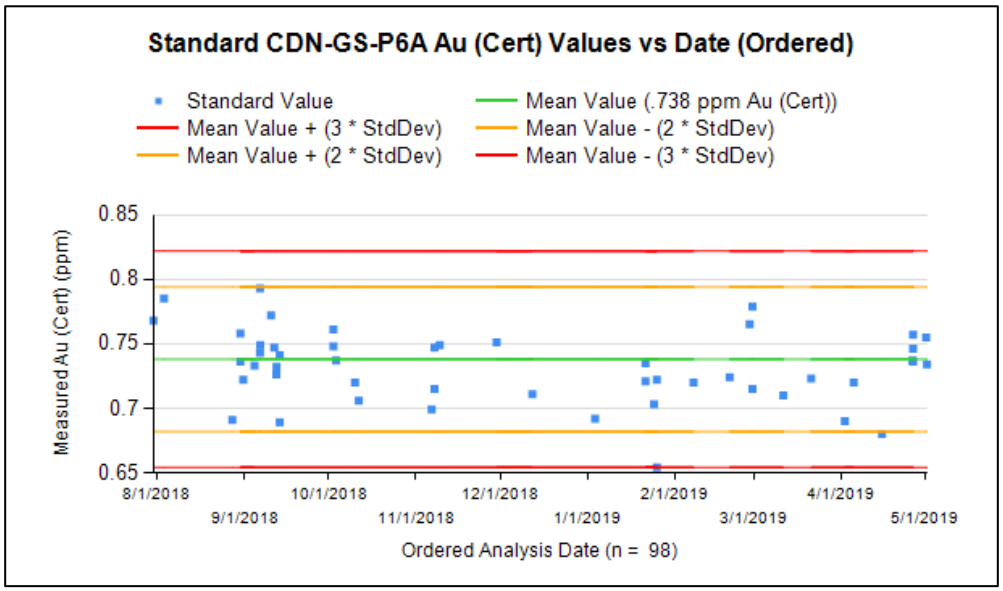
Table 12.1 lists the gold and silver CRMs used by Integra.

Table 12.1 Integra Certified Reference Materials

Reference Material	Certified Value (g Au/t)	2 Std Dev (g Au/t)	Certified Value (g Ag/t)	2 Std Dev (g Ag/t)	No. of AAL Analyses	Metals with Certified Values	Reference Material Composition
CDN-CM-38	0.942	0.072	6.0	0.4	83	Au,Ag,Cu,Mo	K-silicate, silicic and sericitic altered intermediate volcanic and related intrusive rocks exhibiting porphyry-style Cu-Au mineralization. 800 kg of this material was combined with 7 kg of a Au-Cu-Mo concentrate.
CDN-CM-41	1.60	0.15	8	1	30	Au,Ag,Cu	Mineralization from the Minto Mine, Yukon, Canada. Comprised of primary chalcopyrite and bornite pervasively disseminated and as stringers within foliated granodiorite rich in secondary biotite and magnetite. Au is associated with bornite and is rarely observed as free gold.
CDN-GS-P6A	0.738	0.056	81	7	98	Au,Ag	Mineralization from the low-sulfidation Santa Elena property, Mexico, hosted by silcified Tertiary andesite and rhyolite flows. Gangue minerals include quartz, calcite, chlorite and fluorite.
CDN-GS-1P5Q	1.329	0.100	n/a	n/a	49	Au	205 kg of volcanosedimentary mineralization from Amaruq Au project, Canada, blended with 595 kg of granite. Gold mineralization is associated with silicification and/or veins, along with various quantities of pyrrhotite, arsenopyrite, pyrite, and chalcopyrite.
KLEN 73915	1.080	0.03	n/a	n/a	141	Au	sulfide; blended raw materials manufactured to provide a suitable substrate for the analytes of interest.
KLEN 74110	0.237	0.002	n/a	n/a	152	Au	oxide; blended raw materials manufactured to provide a suitable substrate for the analytes of interest.
KLEN 74383	4.93	0.10	47.6	4.8	79	Au,Ag	oxide; blended raw materials manufactured to provide a suitable substrate for the analytes of interest.
KLEN 74589	8.65 (non-grav) 8.49 (grav)	0.36 0.22	4395	215	3	Au,Ag	oxide; blended raw materials manufactured to provide a suitable substrate for the analytes of interest.
OxD108	0.414	0.024	n/a	n/a	34	Au	Pulverized basalt rock and feldspar minerals blended with finely pulverized and screened Au-containing minerals. Oxide
SL77	5.181	0.156	29.1	1.2	18	Au,Ag	Pulverized feldspar minerals, basalt rock and barren iron pyrites were blended with finely pulverized and screened gold and silver-containing minerals.
SN74	8.981	0.222	51.5	1.5	103	Au,Ag	Pulverized feldspar minerals, basalt rock and barren iron pyrites were blended with finely pulverized and screened gold and silver-containing minerals.
SP72	18.16	0.35	83	2.2	10	Au,Ag	Pulverized feldspar minerals, basalt rock and barren iron pyrites were blended with finely pulverized and screened gold and silver-containing minerals.

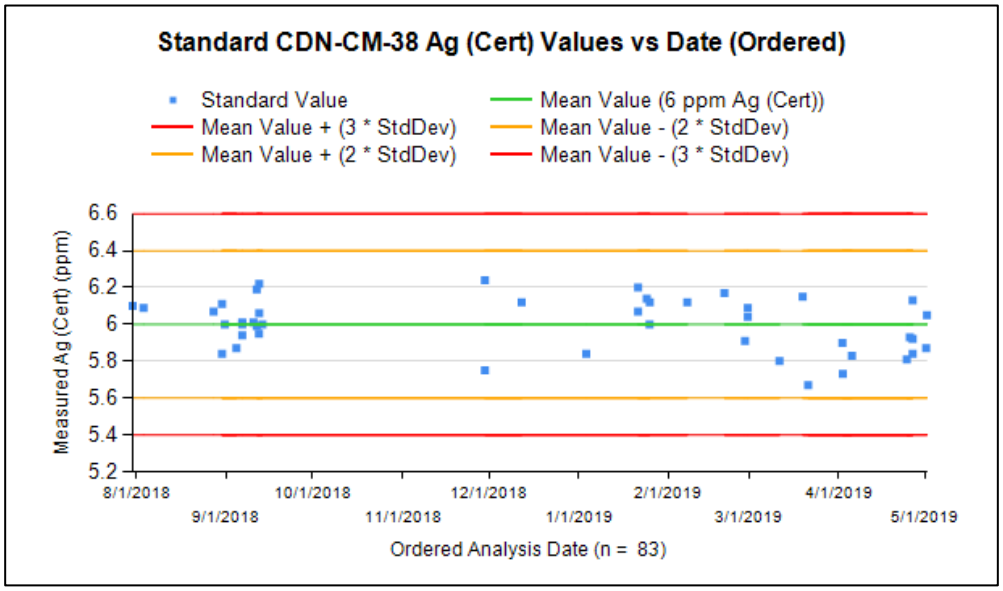
The AAL gold analyses of the Integra CRMs met normal performance thresholds, with a moderate number of 'failures', although gold analyses of many either tended to have a low bias or clearly showed a low bias. Figure 12.5 shows a plot of the AAL gold analyses of CRM CDN-GS-P6A, which has a certified value of 0.738 g Au/t. While none of the analyses are 'failures', there is a clear low bias in the analyses in the time period of the central portion of the plot. This is typical of the AAL gold analyses of most of the CRMs.

Figure 12.5 CRM CDN-GS-P6A Gold Analyses



The AAL silver analyses of the eight CRMs that have certified values returned excellent results, with generally good precision and accuracy, leading to few 'failures' and no bias, except for in the analyses of SN784, which show a high bias although without 'failures'. Figure 12.6 shows typical results for AAL's silver analyses of the CRMs.

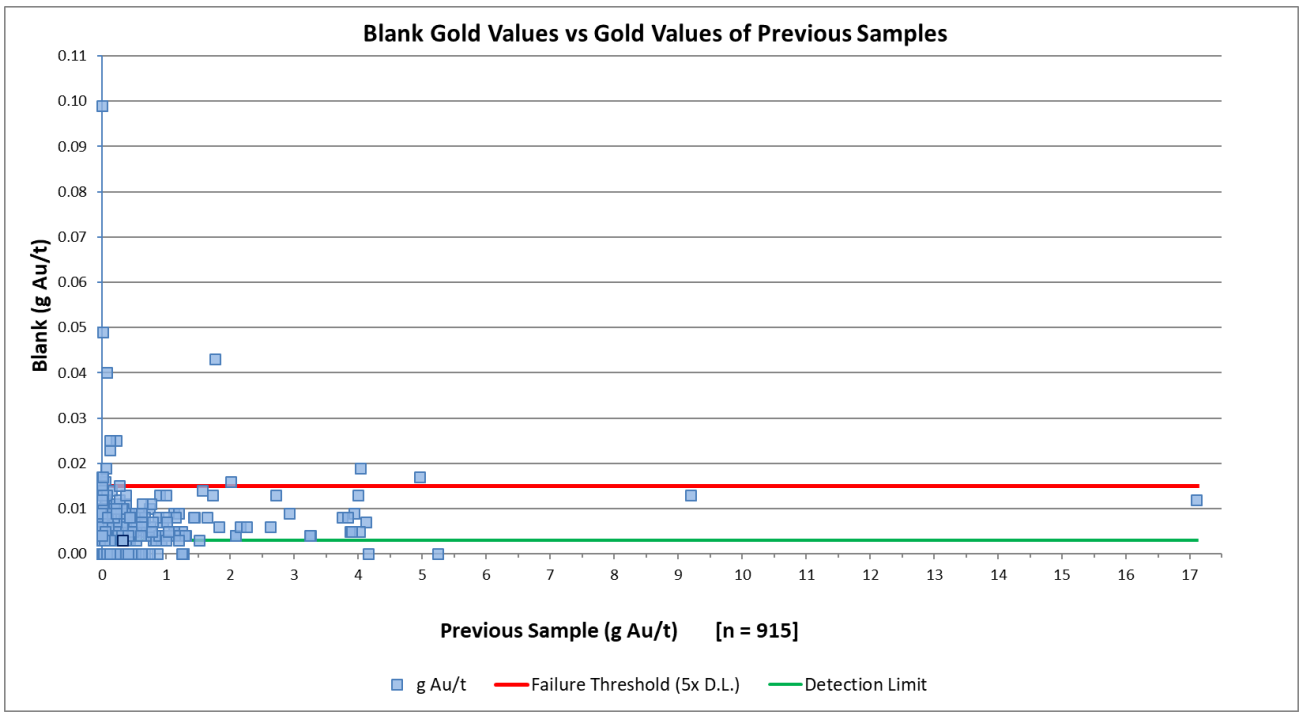
Figure 12.6 CRM SN74 Silver Analyses



Coarse Blanks. Coarse blanks are samples of barren material that are used to detect possible contamination in the laboratory, which is most common during sample preparation stages. In order for analyses of blanks to be meaningful, they must be sufficiently coarse to require the same crushing and pulverizing stages as the drill samples. It is also important for a significant number of the blanks to be placed in the sample stream within, or immediately following, a set of mineralized samples, which would be the source of most contamination issues. In practice, this is much easier to accomplish with core samples than RC.

Blank results that are greater than five times the lower detection limit of the relevant analyses are typically considered failures that require further investigation and possible re-assaying of associated drill samples. The detection limit of the AAL analyses was 0.003 g/t for gold and 0.020 g/t for silver, so blank samples assaying in excess of 0.015 g Au/t and 0.100 g Ag/t are considered to be failures. Figure 12.7 shows a plot of the AAL analyses of the coarse blanks (y-axis) versus the gold values of the previous samples, which would be the likely source of any in-lab contamination.

Figure 12.7 Coarse Blank Gold Values vs. Gold Values of Previous Samples



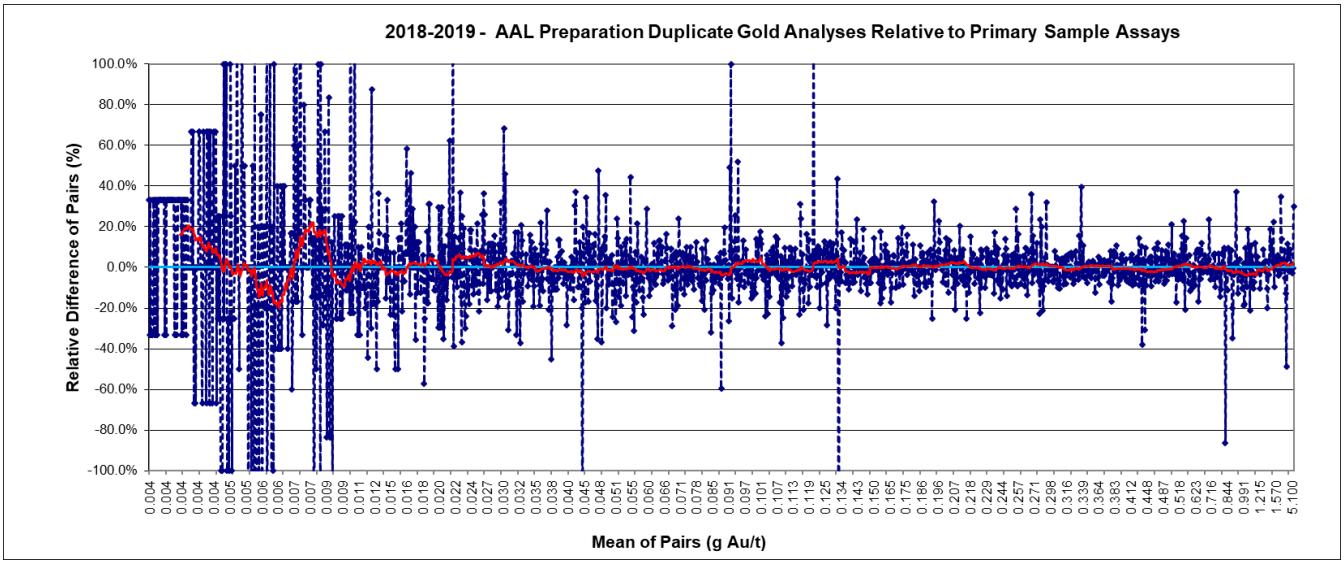
Of the 915 AAL analyses of the blanks, 14 exceed the failure threshold, with values ranging from 0.016 to 0.099 g Au/t. Only the highest value exceeds 0.050 g Au/t, and it is therefore the only potentially material failure.

There are 889 AAL silver analyses of the coarse blanks. Using the reported detection limit of 0.020 g Ag/t, 93% of the AAL analyses of the blanks are technical failures. However, the highest value of the blank analyses is 4.86 g Ag/t, which is not of a magnitude that would be material to the project. Less than 3% of the AAL blank analyses are greater than 1 g Ag/t. Possible explanations for the extreme failure rate include: (i) the coarse blank material was not barren with respect to silver; and (ii) the reported detection limit of the silver analyses is inaccurately low.

RC Field Duplicates. RC field duplicates are secondary splits of original 1.52-meter (five-foot) samples collected at the RC rig simultaneously with the primary sample splits. Field duplicates are used to evaluate the total variability introduced by subsampling, including at the drill rig and in the laboratory (subsampling of the coarse rejects and pulps), as well as the variability in the analyses. Field duplicates should therefore be analyzed by the primary analytical laboratory.

Excluding pairs in which both the RC field duplicate and primary sample assays returned less-than-detection-limit results, there are a total of 1,708 pairs of gold analyses and 2,199 pairs of silver analyses. Figure 12.8 is a relative-difference graph that compares the RC duplicate data to the primary samples.

Figure 12.8 RC Field Duplicate Gold Results Relative to Primary Sample Assays



There is no bias in the data, suggesting that there were no material issues with the drill-site and all additional downstream sample splitting. The mean of the duplicates (0.239 g Au/t) is very close to that of the primary samples (0.242 g Au/t), and the mean of the RDs is 1%. The variability is well within an acceptable range, especially so for an epithermal deposit, with an average AVRD of 14% that includes all data (no outliers were removed), and which decreases at higher grades (e.g., at a 0.2 g Au/t MOP cutoff, the average AVRD is 6%).

The silver field-duplicate data yield very similar results as for gold. There is no bias evident in the relative-difference graph, the means of the silver analyses of the duplicates and original samples are identical, the average RD is -1%, and the average AVRD is 19%, decreasing to 9% at a more relevant MOP cutoff of 15 g Ag/t. No outlier pairs were removed from these statistics.

12.3 Additional Data Verification

In addition to the more structured verification procedures discussed above, extensive verification of the project data, primarily the historical data, was undertaken throughout the process of the resource modeling. The careful work involved in the explicit modeling of the geology and gold and silver sectional modeling led to ad-hoc checking of the accuracy of a variety of data, such as hole locations, hole orientations, drill-hole lithologic attributes, and specific gold and/or silver assays. For example, during Integra's cross-sectional geologic modeling, and MDA's modeling of the mineralization, historical holes were identified as having lithologic and assay information that was strongly at odds with adjacent holes. While paper survey records supported the database locations in some cases, a judgment was made that the hole's location must be inaccurate, and these holes were excluded from use in the resource estimation. Many individual historical assays, as well as assays in entire mineralized intervals, were questioned and then confirmed by paper records and, in some cases, corrected in the project database as a result of working closely with the data during modeling.

The Integra drilling provided another component to the verification of the historical data. Integra's ongoing drill program led to repeated updates to the project databases. After each batch of new drill data was added, the data were compared to the gold, silver, and geological modeling that was already completed and at that point needed updating. In all cases where the Integra drill data extended into modeled areas at both DeLamar and Florida Mountain, the addition of the Integra data did not lead to material changes to the style of mineralization as indicated by the historical data, nor the magnitude of the gold and silver mineralization. However, the Integra drilling did succeed in substantially extending the limits of the mineralization defined by the historical data in several areas. This detailed work with the Integra drill data in the context of the historical information played a critical role in the validation of the historical data.

12.4 Site Inspection

Mr. Weiss visited the project site for three days, on August 1 – 3, 2017, accompanied and assisted by Ms. Kim Richardson of Jordan Valley, Oregon. Ms. Richardson is a geologist who joined the DeLamar mine staff in 1980 and eventually held the positions of Senior Mine Geologist, Mine Superintendent, and Mine General Manager before leaving the project in 1997. Mr. Weiss reviewed the property geology, exposures of mineralized rocks within and near the still accessible open pits, and areas of historical exploration drilling peripheral to the open pits, in both the DeLamar and Florida Mountain areas. Historical exploration data on file at the DeLamar mine-site office was reviewed, including geologic maps and cross sections from various areas, mainly dating to the late 1980s.

Mr. Weiss attempted to verify historical drill-hole collar locations peripheral to the open pits. Nearly all historical drill sites external to the pits and waste dumps have undergone reclamation since closure activities began in 2003. Seven drill collars were found in the Sullivan Gulch and Ohio areas. Metal tags marked with the hole numbers were found at a few of the collars, but none of these were legible. Nevertheless, the eight collar locations were recorded with a hand-held Garmin GPS-62 receiver in UTM WGS84 projection in case that the holes can be identified in the future.

Mr. Gustin visited the project site on October 16 through October 18, 2018. All principal areas of mineralization at the DeLamar and Florida Mountain areas were visited in the field, as well as exploration areas both on the project (Town Road – Henrietta) and north of the project. Numerous altered and mineralized areas throughout the project and adjacent areas were visited, open-pit walls were examined, and mineralized intervals from multiple core holes were closely inspected. An actively drilling and sampling RC rig was also visited, and all project procedures related to the RC and core drilling programs, data collection, and data storage were reviewed. Where appropriate, recommendations were provided to the Integra technical team.

12.5 Independent Verification of Mineralization

No samples were collected from the DeLamar project for verification purposes by the authors. Gold and silver production from the historical underground mines and more recent open-pit operations is well documented in both private historical records and publicly documents. In the opinion of Mr. Gustin and Mr. Weiss, independent sampling for the purposes of verifying the DeLamar and Florida Mountain mineralization is not needed.

12.6 Summary Statement

The authors experienced no limitations with respect to data verification activities related to the DeLamar project. In consideration of the information summarized in this and other sections of this report, the authors have verified that the DeLamar project data are acceptable as used in this report, most significantly to support the estimation and classification of the mineral resources reported herein. This conclusion is further supported by the fact that: (*i*) the historical drill data have undergone an extensive amount of validation by both the authors and Integra; and (*ii*) the historical drilling data formed the basis of a commercial mine that operated successfully over an extended time period.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section, prepared under the supervision of Mr. Jack S. McPartland, Senior Metallurgist with McClelland, summarizes the metallurgical testing conducted on samples from the DeLamar and Florida Mountain areas, and historical processing of materials from the same two areas. Estimates of recovery and reagent consumption developed for the processing methods selected for the PEA are included. Mr. McPartland has reviewed the information in this section and believes it is a reasonable summary of the mineral processing, metal recoveries, and metallurgical testing for the DeLamar project as presently understood. The terms "ore" and "whole-ore" used in this section refer to material tested or to be potentially processed and do not imply economic material.

Metallurgical testing is considered in two parts; historical (pre-1990) and current (2018-2019) testing. The current metallurgical testing forms the primary basis for the recovery and reagent estimates used in this study. The samples used for the current testing generally come from the current resources and are believed to be reasonably representative of the material considered for processing in the PEA. A review of pre-1990 metallurgical testing and processing is presented and contributes to an understanding of the mineralogy and metallurgy for the project. As records of the sample sources for the historical work are incomplete, and in some or most cases the material represented by those samples was likely processed during earlier commercial operations, data from the historical testing are of more limited use than data from the ongoing 2018-2019 metallurgical testing.

Nearly all of the historical metallurgical tests and processing data summarized below were originally reported in Imperial units, but in some cases metric weights were reported that were mixed with Imperial distance and concentration units. Use of the original reported units is retained in parts of this section for historical clarity and to avoid awkwardness; the reader is referred to Section 2.2 for the appropriate conversion factors.

13.1 DeLamar Area Production 1977 – 1992

13.1.1 DeLamar Area Mill Production 1977 - 1992

Useful information with respect to mineral processing of DeLamar area gold-silver mineralization by milling and subsequent cyanide leaching is derived from mill production records from the historical open-pit mining operations from 1977 through to the end of 1992. All ore during this time period was mined from the DeLamar area and was processed by crushing, grinding, and cyanide leaching, followed by precipitation with zinc dust and in-house smelting of the precipitate to produce silver-gold doré. Elkin (1993) estimated the doré to contain 89% silver and 2% gold. After leaching, the solids were concentrated in a series of five thickening tanks and then pumped to a tailings impoundment. During mine closure the tailings were partially dewatered and capped with layers of clay and soil as part of the mine reclamation program.

The DeLamar area produced 421,300 ounces of gold and about 26 million ounces of silver from 1977 through 1992 from 11.686 million tonnes of ore processed with average mill head grades of 1.17 grams Au/t and 87.1 grams Ag/t (Elkin, 1993). The data from Elkin (1993) presented in Table 6.1 demonstrate mill recoveries during the first 15 years of mine operation averaged 96.2% for gold and 79.5% for silver.

It should be noted that Elkin (1993) surmised that, "*Based on historical records and laboratory testing, the metallurgical recovery of gold is projected to be about 94 percent and 77 percent for silver.*"

In 2017, Integra compiled milled silver and gold grades and recoveries from available DeLamar mine monthly company production reports. Records were found for 51 months from December of 1981 through September of 1998. Using quarterly production data for royalties, Integra also compiled information on the specific open pits that provided the mill feed for the monthly mill production and recovery estimates. With the assistance of Ms. Kim Richardson, and based on her knowledge of the specific pits that provided the mill feed through time, Integra estimated the proportions of unoxidized-type ore versus oxidized ore, that were processed by the mill for the 51 months of compiled mill recovery information. This analysis indicated variations in monthly mill recovery rates of from 76% to 98% gold, and from 67% to 87% silver. This included nine months where the mill feed was estimated to contain as much as 50% or more unoxidized ore. This may give a qualitative indication of the variation in gold and silver recoveries obtained generally by grind-leach with the estimated proportions of oxidized-type ore feed, although other variables may have contributed to the recoveries reported.

It should be noted here that agitated leach (cyanidation) testing conducted by McClelland on samples from Integra's 2018 and 2019 drill samples (discussed in following subsections) indicate that gold extractions by grind-leach from the unoxidized-type (alternatively reduced type) material are generally poor but highly variable, and the reasons for the variability are poorly understood.

13.1.2 Cyanide Heap Leaching 1987 - 1990

NERCO constructed a cyanide heap-leach pad, which was in operation for the last quarter of 1987 until the final quarter of 1990, using low-grade run-of-mine material dumped by truck and ripped to provide permeability. The material size was reported to be approximately 70% at >20 centimeters (>8 inch).

The heap-leach production and recovery from this pad are summarized in Table 13.1. The pad base and subsequent stacked material became unstable and began to collapse in mid-1990. Quarterly production records indicate no material was placed on the heap after the second quarter of 1990. In early 1991, the entire heap was removed and placed into the tailings facility.

Table 13.1 1987 – 1990 Heap Leach Summary
(mine records from Integra, 2017)

Heap Leach Q4 1987 - Q4 1990			
		Ag g/tonne	Au g/tonne
Total Stacked (tonnes)	2,344,037	31.78	0.41
Contained Ounces Stacked		2,227,571	28,836
Recovered Ounces		173,281.00	11,683.00
Heap Leach Recovery		**8%**	**41%**

The incomplete leaching that likely resulted from the pad failure would have adversely affected the reported heap-leach recoveries. These recoveries are not believed to be indicative of recoveries expected from heap leaching of DeLamar oxide and transitional materials.

13.2 Mineralogy from Metallurgical Studies

As reported by Perry (1971), Hazen Research Inc. ("Hazen") in Golden, Colorado undertook a detailed petrographic and mineralogical study of four sections of drill core from the DeLamar area in 1971. The host rock was described as highly altered porphyritic rhyolite, initially altered to sericite and kaolinite and then silicified and cut by numerous quartz veinlets. Naumannite (Ag_2Se) was identified as the chief primary silver-bearing phase, accounting for 75-78% of the total silver present. Argentite (Ag_2S) was the other primary silver mineral, accounting for 15-20% of the total silver. Minor gold was found in quartz gangue and as intergrowths in naumannite. The core samples were reported to be at least partly oxidized to a depth of 51.8 meters. Secondary silver minerals in the oxidized portions of core included the silver halide cerargyrite (AgCl), native silver, and argentojarosite, but these minerals together account for only a small fraction of the total silver in the samples. Approximately 65% of the naumannite and argentite was reportedly less than 62 microns in diameter; the coarser grains were typically found within quartz veinlets. Sulfide and native metal phases identified in the heavy media concentrates included:

Pyrite: present as disseminated grains in the altered volcanic rocks and within quartz veins where it was coarser grained and occurred as malnikovite, a fine-grained black pyrite phase initially deposited as a colloidal pyrite gel. Minor marcasite was present as encrustations on malnikovite and as scattered crystals in quartz veinlets. Only rarely were silver minerals directly associated with, or intergrown with, pyrite and marcasite.

Argentite: as coarse grains and tabular masses up to 2 millimeters in diameter along fractures and veinlets, and more commonly as extremely fine sized anhedral grains disseminated within quartz gangue.

Naumannite: as fine-grained, generally anhedral grains disseminated in quartz and along fractures. Less commonly it forms crystals and crystal aggregates on top of quartz crystals lining vugs at the center of veinlets.

Cerargyrite: as thin coatings and sheets along fractures, and as globular or spherical grains that have replaced naumannite in open vugs. It also rarely occurs as clear yellow to chartreuse dodecahedral crystals in quartz vugs.

Native gold: in trace amounts as small anhedral blebs in quartz or in naumannite. The gold blebs rarely exceed a few microns in size.

Electrum: as pale yellow, 5 micron or smaller blebs within cerargyrite after naumannite.

Native silver: as a secondary mineral occurring as tabular sheets and masses along fractures and veinlets; grains rarely exceed 2 millimeters in diameter.

Chalcopyrite: present in trace amounts, rarely associated with pyrite and more commonly as euhedral blebs within partly replaced naumannite.

Traces of galena were noted, occurring as small isolated euhedral grains, and the lead selenide clausthalite (PbSe) was suspected to be present. Very minor native selenium was identified by X-ray powder diffraction and was interpreted as a byproduct of naumannite alteration during the oxidation process. Other oxidation products including jarosite, argentojarosite, psilomelane, geothite, and lepidocrocite were identified.

In 1978, a mineralogical study was conducted by Newmont Exploration Limited (Ahlrichs, 1978) to evaluate improved silver recoveries from Glen Silver feed to the DeLamar processing plant. Samples of feed and plant tailings were evaluated by assay, microscopic examination, XRF, XRD and electron microprobe analysis. There was no information concerning the oxidation classification of the material studied (oxide, transitional or unoxidized). The leached tailings contained approximately 0.5 oz Ag/ton. It was observed that about 90% of the silver was in the form of naumannite (Ag_2Se). Remaining silver was present as acanthite (Ag_2S). Very little of the silver contained in the tailings was free. Greater than 85% of the unrecovered silver in the tailings occurred as fine inclusions (1 – 26µm) in quartz. It was concluded that no significant improvement in silver recovery would be gained by finer grinding.

In 1982, a mineralogy study was conducted by Hazen on eight drill core interval samples from four Glen Silver drill holes (designated GS-1, GS-2, GS-3 and GS-4), to evaluate the mode of occurrence of silver in the samples. The samples were reported to material from oxidized zones (6 samples), a partially oxidized zone (1 sample) and an unoxidized zone (1 sample). Silver deportment was evaluated using optical and scanning electron microscopy ("SEM") analysis. The silver mineralization was identified in the oxide samples as silver sulfide, probably acanthite, containing 90% silver and 10% sulfur. The acanthite was associated, mostly as inclusions, with goethite, which was derived from the oxidation of pyrite. Silver minerals were not clearly identified in either the partially oxidized or unoxidized samples, and it was speculated that the contained silver might be present in solid solution with pyrite. Considering the fine grain size of the acanthite and its common occurrence as inclusions in goethite, it was speculated that the material represented by the samples might be refractory to cyanidation, with respect to silver recovery.

13.3 Historical Metallurgical Testing

Multiple metallurgical testing programs were conducted during the 1970s and 1980s on samples from the DeLamar and Florida Mountain deposits. These studies were commissioned by Earth Resources (1970s) and NERCO (1980s). The Earth Resources studies conducted during the 1970s were focused on milling and whole ore agitated cyanidation leaching of the various material types. The NERCO work in the 1980s was more focused on heap leaching of the various material types. A summary list of the studies is presented in Table 13.2.

Table 13.2 Historical Mineralogy and Metallurgical Testing, DeLamar and Florida Mountain Deposits

Reference	Company		Sample			Type of Testing
	Commissioned by	Laboratory	Source	Type	Number	
Perry (1971)	Earth Resources	Hazen	DeLamar - Sommercamp	Drill Core	3 drill holes	Mineralogy
Miyoshi et al. (1971)	Earth Resources	Hazen	DeLamar - Sommercamp	Drill Core	3 Comps.	Flotation, Agitated Cyanidation Salt Roast - Acid Brine Leach Zn Precipitation
Miyoshi et al. (1974)	Earth Resources	Hazen	DeLamar - North DeLamar	Drill Cuttings	1 Comp.	Flotation, Agitated Cyanidation Salt Roast - Cyanide Leach De-Sliming, Thickening
Miyoshi et al. (1974)	Earth Resources	Hazen	DeLamar - North DeLamar	Drill Core	1 Comp.	Flotation, Agitated Cyanidation Salt Roast - Cyanide Leach De-Sliming, Thickening
Ahlrichs (1978)	Unknown	Newmont Exploration	DeLamar	Plant Feed and Tails	2	Mineralogy
Schmidt (1982)	Mapco Minerals Corp.	Hazen Research	DeLamar - Glen Silver	Drill Core	8	Silver Deportment Mineralogy
Nerco Internal Memo (1986)	Nerco Minerals	Nerco Minerals	DeLamar - Glen Silver North DeLamar	Bulk	7	Column Tests Coarse Bottle Roll Tests Grind - Agitated Cyanide Leach
Rak et al. (1989)	Nerco Minerals	Hazen	Sullivan Gulch	Bulk (?)	1	Mineralogy, Gravity, Flotation, Agitated Cyanidation (whole feed, Grav. and Flot. Conc. and Tails)
Kilborn (1988) Hampton (1988) Satter (1989)	Nerco Minerals	Nerco Minerals	Florida Mountain - Stone Canyon, Sullivan and Clark	Drill Core	4 Comps.	Column Tests Vat Leach Tests Agitated Cyanide Leach Ball & Rod Mill Work Indices Thickening
Satter (1989)	Nerco Minerals	Nerco Minerals	Florida Mountain - Stone Cabin	Bulk	1	Pilot Column Test ("Run-of-Dump" material)

13.3.1 1970s Earth Resources – Hazen Testwork

A series of metallurgical testing programs was conducted on drill hole samples from the DeLamar Sommercamp and North DeLamar areas in the 1970s. A summary of representative flotation test results and agitated cyanide leach test results from each of the three testing programs are presented in Table 13.3 and Table 13.4, respectively.

Table 13.3 Summary Results, Rougher Flotation Tests, Hazen 1970 Studies

Sample		Feed Size	Concentrate			Head Grade		Recovery	
			Weight,	Grade					
No.	Description	Mesh	%	opt Au	opt Ag	opt Au	opt Ag	% Au	% Ag
Comp. C[1)]	Sommercamp	100	32.3	0.053	19.0	0.024	6.46	71.7	93.3
HRI 6233[2)]	N. DeLamar	-100	33.8	0.174	5.8	0.072	2.61	81.7	75.1
N/A[3)]	N. DeLamar	200	30.0	0.063	7.6	0.026	2.77	73.2	83.3

1) Miyoshi et al. (1971);

2) Miyoshi et al. (Jan. 1974)

3) Miyoshi et al. (June 1974)

Table 13.4 Summary Results, Agitated Cyanidation Tests, Hazen 1970 Studies

Sample		Feed Size	Leach Time	Recovery		Head Grade		NaCN	CaO
No.	Description	Mesh	hours	% Au	% Ag	opt Au	opt Ag	lb/ton	lb/ton
Comp. C[1)]	Sommercamp	-48	72	75.0	96.0	0.02	6.36	3.8	12
HRI 6233[2)]	N. DeLamar	-48	48	84.1	>88.9	0.09	2.58	8.84	12
N/A[3)]	N. DeLamar	65	72	95.7	91.9	0.023	3.35	6.02	1.2

1) Miyoshi et al. (1971)

2) Miyoshi et al. (Jan. 1974)

3) Miyoshi et al. (June 1974)

Hazen carried out initial metallurgical studies of three composites from drill core partly used in Hazen's 1971 mineralogy studies, as reported by Miyoshi et al. (1971). This material was later described (Myoshi and Light, 1974) as being from the Sommercamp area. The composites tested ranged in grade from 0.69 to 1.03 g Au/t and from 218.1 to 266.1 g Ag/t. Metallurgical tests included flotation, cyanidation, and a salt roast followed by acid-brine leach. Representative flotation and cyanidation results are shown in Table 13.3 and Table 13.4.

In 1974, Earth Resources commissioned Hazen to perform metallurgical studies on conventional-rotary drill cuttings from the North DeLamar zone (Miyoshi, 1974a), to determine similarities and differences to the Sommercamp mineralization reported by Perry (1971). A single composite sample, designated HRI 6233 was prepared from 41 intervals of conventional-rotary drill cuttings from eight North DeLamar drill holes. Testing included agitated cyanidation tests (11), evaluation of a salt roast-cyanide leach procedure, de-sliming, flotation testing and thickening testing. Test results indicated better recoveries by cyanidation than by flotation. High cyanide consumption was noted.

In 1974, Earth Resources also commissioned Hazen to conduct further metallurgical testwork on drill core from the North DeLamar area (Miyoshi, 1974b). Hazen conducted mineralogical studies, Bond ball mill work index, specific gravity and bulk density measurements, flotation with deslime tests and screen analysis, and cyanide-leach tests at various levels of grind, as well as filtration and thickening tests at various flocculent levels, and carbon-in-pulp and zinc-precipitation tests. The composite tested was described as porphyritic rhyolite or quartz latite with argillic alteration, containing about 10% clay. The Bond ball mill work index of the sample was determined to be 15.5 kW-hr/ton.

13.3.2 1989 Sullivan Gulch Testing for NERCO

A metallurgical investigation was conducted for NERCO by Hazen on a sample of Sullivan Gulch material. Testing included mineralogical characterization, and evaluation of gravity, flotation, cyanide leaching (both whole feed gravity/flotation products) and pressure oxidation treatment of flotation concentrate.

The sample grade was 1.06 g Au/t, 72.0 g Ag/t and 2.64% sulfide sulfur. Mineralogical examination showed that approximately 10% of the sample was sulfide mineral, with pyrite being the dominant

sulfide. Gold was present in free native electrum, as coarse as 50 to 100µm, and as coarse intergrowths in pyrite. Silver occurred mainly as electrum and argentite, in mostly liberated forms.

The study concluded that multiple processing routes had merit for treating the Sullivan Gulch material, but that no single processing technique was effective for attaining 90% recovery of both gold and silver. Hazen found that gold and silver extractions in excess of 90% could be achieved on the unoxidized samples from Sullivan Gulch using a combination of gravity separation, followed by additional grinding and a second stage of flotation followed by agitation cyanide-leach on the gravity tails (Rak et al., 1989). This combined recovery does not account for gold and silver losses that would be incurred during subsequent processing of the gravity and flotation concentrates (combined mass 15.69% of feed weight) for gold and silver recovery. Representative results for gravity, flotation and cyanide leaching alone, along with combinations of these processes considering both flotation concentrate atmospheric leach and leach after pressure oxidation ("POX") treatment are presented in Table 13.5.

Table 13.5 Summary Results of Hazen Testing, Sullivan Gulch Sample
(from Rak et al., 1989)

Process	Test No.	Grind Size, Mesh	Concentrate Product	Concentrate Wt %	Extraction % Au	Extraction % Ag	Tail Assay Au oz/ton	Tail Assay Ag oz/ton
Gravity Alone	T-3	95%-200	Gravity Concentrate	6.72	**70.4**	**68.0**	0.020	0.99
Flotation Alone	F-4	67%-200	Flotation Concentrate	17.77	**77.1**	**98.1**	0.013	0.05
Cyanidation	AL-7	90%-500	CN Leach	N/A	**73.5**	**80.2**	0.008	0.42
Gravity, regrind/CN leach of gravity tails	T-3G	36%-200	Gravity Concentrate	6.72	70.4	68.0		
		85%-500	Tailings Regrind/Leach	93.28	20.2	23.1	0.007	0.24
			Gravity Concentrate + Tails Leach	**N/A**	**90.6**	**91.1**		
Gravity, float of gravity tails, CN leach of float tails	T-2GF	95%-200	Gravity Concentrate	5.54	48.0	47.4		
			Flotation Concentrate	10.15	35.2	49.0		
			Tailings Leach	84.31	8.2	3.3	0.003	0.01
			Grav. and Flot. Conc. + Flot. Tails Leach	**N/A**	**91.4**	**99.7**		
Gravity, float of gravity tails, CN leach of reground float conc	T-4GF/ AL-10	95%-200	Gravity Concentrate	1.71	29.5	21.9		
			Flotation Concentrate	9.28	53.3	74.1	0.010	0.12
		56%-400	Reground Flot. Conc. Regrind/CN	9.28	35.9	69.9	0.082	1.36
			Grav. Conc. + Flot. Conc. Regrind/CN	**N/A**	**65.4**	**91.8**		
Gravity, float of gravity tails, POX/CN leach of reground float conc	T-4GF/ AL-11	95%-200	Gravity Concentrate	1.71	29.1	21.9		
			Flotation Concentrate	9.28	53.3	74.1	0.010	0.12
			Flot. Conc. POX/CIL (no regrind)	9.28	52.3	0.3	0.006	23.1
			Grav. Conc. + Flot. Conc. POX/CIL	**N/A**	**81.4**	**22.2**		

Note: PR denotes porphorytic ryolite (sic), QL denotes quartz latite, SC denotes Sommer Camp, Ttb-B denotes Ttb-brecciated, ND denotes North DeLamar,; Tpr-S & B denotes Tpr-silicified & brecciated.

13.3.3 1980s Florida Mountain Testing for NERCO

During the 1980s NERCO conducted column-leach tests using mineralized material from Florida Mountain. No information concerning the oxidation state of the samples tested was available. Statter (1989) summarized metallurgical test work conducted at the DeLamar mine laboratory with Florida Mountain mineralized material in an internal NERCO report. Column-leach and agitation-leach results

were reported for Sullivan drill core, Stone Cabin core, and Clark core. In this case, Sullivan core refers to drill core from the Sullivan claim at Florida Mountain, not the Sullivan Gulch area. The results of the column-leach tests, which were run for approximately 60 days with crush sizes of one-inch and 0.5-inch, are summarized in Table 13.6. The authors are unaware of the column diameter(s) or the oxidation state of the material tested.

Table 13.6 NERCO Florida Mountain Column-Leach Tests
(from Statter, 1989)

Florida Mountain Area	Crush Size	Calc. Head Grade		Reagents lb/ton		Metal Extraction %	
	(inches)	Ag oz/ton	Au oz/ton	NaCN	Lime	Ag	Au
Sullivan	-1	0.248	0.017	2.2	6	41.9	82.3
Sullivan	-1/2	0.227	0.018	2.2	6	53.8	82
Stone Cabin LG	-1	0.255	0.009	1.8	5.2	45.2	85.1
Stone Cabin LG	-1/2	0.317	0.01	1.9	5.2	43.1	84.5
Stone Cabin HG	-1	0.455	0.047	2.1	5.2	39.3	78.1
Stone Cabin HG	-1/2	0.42	0.043	2.3	5.3	47.6	84.3
Clark LG	-1	0.144	0.007	1.8	5.3	37.5	52
Clark LG	-1/2	0.127	0.007	2.2	5.4	53.6	83.7
Clark HG	-1	0.413	0.025	2.2	5.4	36.4	38.7
Clark HG	-1/2	0.446	0.023	2	5.3	48.9	59.3

Additional column-leach tests were conducted by NERCO in 1988 at the DeLamar mine laboratory with drill-core and mine-dump samples from the Stone Cabin area, and samples from a trench at the Tip Top area (Kilborn, 1988; Hampton, 1988). The tests were run for 56 and 60 days with material crushed from 70% at minus 0.25 inches, to minus two inches, as summarized in Table 13.7. Mr. McPartland is unaware of the column diameter(s) or the oxidation state of the material tested.

Table 13.7 Other NERCO Florida Mountain Column-Leach Tests
(from Hampton, 1988 compiled by Integra, 2017)

Area / Type	Crush Size	Caluclated Head Grade		Duration	Adjusted* Metal Extraction	
		Ag oz/ton	Au oz/ton	days	Ag %	Au %
Stone Cabin Dump	1"	1.761	0.108	60 days	39.7	83.1
Stone Cabin Core	-1"	0.514	0.019	60 days	31.5	92.2
Stone Cabin Core	- 1/2"	0.466	0.018	60 days	42.9	92.6
Stone Cabin Core	-1/2"	0.53	0.35	60 days	36.2	78
Tip Top Trench	-2"	0.506	0.03	56 days	41.6	92.2
Tip Top Trench	-1"	0.576	0.032	56 days	42.8	91.5
Tip Top Trench	70% -1/4"	0.636	0.03	56 days	45	95

** denotes an internal DeLamar mine assay factor was applied to silver and gold analyses.*

Also, Statter (1989) reported a pilot column-leach test was performed in 1988 or 1989 using 14,850 pounds of Stone Cabin "run of dump" material. The test was likely conducted at the DeLamar mine laboratory. Leaching was conducted for 63 days resulting in 15.8% silver recovery and 72.2% gold recovery (Statter, 1989).

13.4 Integra 2018-2019 Metallurgical Tests

A metallurgical testing program was initiated at McClelland by Integra in September 2018, with the primary objectives of evaluating and optimizing processing options for the various material types from both the DeLamar and Florida Mountain deposits. That testing program is ongoing, with testing expected to continue beyond the end of 2019. Results from completed testing were summarized in three update reports (McPartland, 2019a, 2019b and 2019c).

McClelland maintains ISO accreditation (ISO/IEC 17025:2005) for analytical services, including fire assay, geochemical assay, carbon and sulfur analyses and solution analyses (including gold and silver) presented in this section of the report. All solution analyses, fire assays and geochemical assays conducted as part of the 2018-2019 McClelland metallurgical testing program were conducted following generally accepted industry practices related to quality control and quality assurance, including the use of blanks and standards in each analytical batch.

Samples used for the 2018-2019 testing were selected to represent the current resources, and information from this testing campaign forms the primary basis for recovery process selection, and estimates of metal recovery and reagent consumptions, for materials from the DeLamar and Florida Mountain mineral resources. Testing has included evaluation of cyanide heap leaching, grind plus agitated leaching, gravity concentration, flotation, and flotation concentrate regrind with agitated leaching. Available testing results have shown that the oxide and transitional material types from both the DeLamar and Florida Mountain deposits generally will be amenable to heap-leach cyanidation treatment. Higher gold and silver recoveries are indicated for these materials by grind plus agitated leach.

Unoxidized materials from the Florida Mountain deposit are consistently amenable to grind plus agitated leach and respond very well to upgrading by gravity and flotation methods. Flotation concentrate generated from the Florida Mountain unoxidized material is readily amenable to regrind plus agitated leach processing.

Unoxidized materials from the DeLamar deposit generally respond well to upgrading by gravity and flotation methods. The majority of unoxidized material types in the 2018 and 2019 testwork from the DeLamar deposit are not readily amenable to grind plus agitated leach. Further testwork on agitated leaching on DeLamar materials will be carried out in 2019 and 2020. For the DeLamar unoxidized material types not amenable to grind-leach, the most likely processing option will include milling followed by flotation (possibly also with gravity concentration) to produce a concentrate. Possible processing options for gold and silver recovery from the concentrate include shipment off site for toll processing; regrind followed by agitated cyanidation; or on-site oxidative treatment (such as pressure oxidation, roasting, or bio-oxidation), followed by agitated cyanidation of the oxidized concentrate. Some DeLamar unoxidized material can be processed by grind plus agitated cyanide leach. Further testing and geological modelling will be required to evaluate the feasibility of such approaches for commercially processing select DeLamar unoxidized material.

Samples used for testing include a total of 185 drill-core composites and four bulk samples. Drill-core composites were prepared from a total of 36 drill holes (29 holes from DeLamar and 8 holes from Florida Mountain). Composites were prepared considering area, oxidation, depth, lithology, alteration,

grade and grade continuity. The four bulk samples were taken from the DeLamar area, and represent oxide material (1 sample) and transitional material (3 samples).

The scope of testing conducted on individual samples generally depended on the oxidation classification. All samples are subjected to a detailed series of head analyses, including fire assay, cyanide solubility analysis, carbon and sulfur speciation analysis, and a multi-element ICP analysis. Metallic-screen fire assays were also conducted on most of the Florida Mountain composites.

13.4.1 DeLamar Area Testing 2018-2019

A summary of the 2018-2019 DeLamar metallurgical samples tested to date at McClelland is shown in Table 13.8.

Table 13.8 Drill Hole Composite Summary, DeLamar 2018-2019 Testing

Area	Drill Hole	Interval meters [1] from	to	Composites Oxide	Transitional	Un-Oxidized	Mixed[2]
DeLamar	IDM18_025	28.6	33.5	0	1	0	0
DeLamar	IDM18_039	55.2	74.7	0	0	1	0
DeLamar North	IDM18_017	19.5	27.1	0	1	0	0
DeLamar North	IDM18_027	15.2	60.7	0	0	0	2
DeLamar North	IDM18_028	23.3	57.6	0	1	1	1
Sommercamp	IDM18_029	19.8	213.4	1	0	5	0
Sommercamp	IDM19_116	0.0	229.2	1	0	4	1
Sommercamp	IDM19_134	26.5	137.5	3	0	1	0
Glen Silver	IDM18_009	18.3	164.6	3	0	2	2
Glen Silver	IDM18_013[3]	54.6	72.1	0	0	1	0
Glen Silver	IDM18_023	41.5	127.6	1	0	4	0
Glen Silver	IDM18_030	60.4	155.9	0	1	2	0
Glen Silver	IDM19_117	31.1	190.0	2	1	4	0
Glen Silver	IDM19_131	15.1	128.3	3	0	2	1
Glen Silver	IDM19_132	107.0	113.1	0	0	1	0
Glen Silver	IDM19_133	34.1	63.1	0	0	2	0
Sullivan Gulch	IDM18_005	93.0	274.3	0	1	8	2
Sullivan Gulch	IDM18_007	185.9	335.3	0	0	8	0
Sullivan Gulch	IDM18_008	71.6	265.2	0	0	5	0
Sullivan Gulch	IDM18_011	71.6	266.7	0	0	8	0
Sullivan Gulch	IDM18_012	301.8	359.7	0	0	2	0
Sullivan Gulch	IDM18_014	166.1	396.2	0	0	13	0
Sullivan Gulch	IDM18_046	275.8	381.0	0	0	8	0
Sullivan Gulch	IDM18_047	263.7	379.5	0	0	4	0
Sullivan Gulch	IDM18_048	115.8	428.2	0	0	11	0
Sullivan Gulch	IDM18_052	207.0	342.7	0	0	5	0
Sullivan Gulch	IDM18_055	190.0	204.2	0	0	1	0
Sullivan Gulch North	IDM18_053	109.4	118.6	0	0	1	0
Sullivan Gulch North	IDM18_054	80.3	88.1	0	0	1	0
DeLamar Deposit Total				**14**	**6**	**105**	**9**

Composites from the DeLamar deposit included 52 from the Sullivan Gulch area, 32 from the Glen Silver area, eight from the Delamar/DeLamar North area and 16 from the Sommercamp area. Of those composites, 14 were classified as oxide, six as transitional and 105 as unoxidized. An additional nine composites were mixed (contained more than one oxidation class). Composites were also classified and grouped according to depth, lithology, alteration and grade.

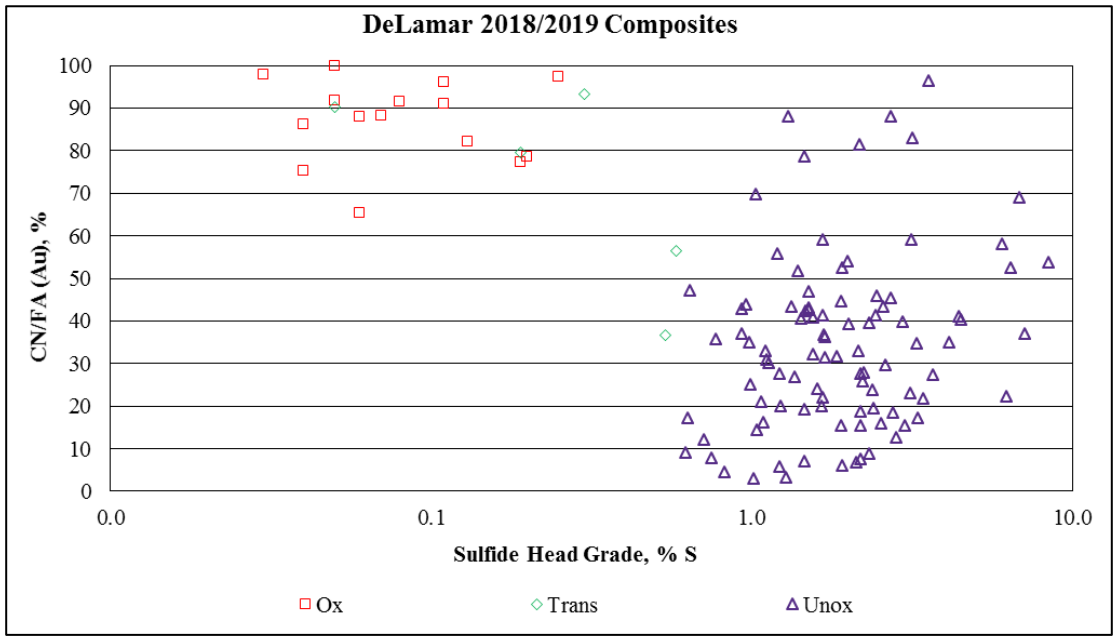
The four bulk samples submitted for testing each weighed about three tonnes and were approximately -350mm (-14 inch) in size. These samples were excavated from the DeLamar area in March 2019 for heap leach testing (Jordan, 2019). The bulk samples included one oxide material sample (sample 4307-

B) and three transitional material samples (samples 4307-A, C and D). The transitional material samples were described as either "Trans Clay" (4307-A) or "Trans Hard" (4307-C and D). The "Trans Clay" sample was selected to represent material with an elevated clay content (determined visually). A detailed description of the sampling procedures and locations was presented in Jordan (2019).

Gold head assay results showed that the DeLamar core composites ranged in grade from 0.19 to 12.63 g Au/t. Silver head grades ranged from 2 to 487 g Ag/t. Average gold head grades for the oxide, transitional and unoxidized composites were 0.47, 0.41 and 1.07 g Au/t, respectively. Average respective silver head grades were 24, 32 and 49 g Ag/t. Average respective sulfide sulfur head grades were 0.2813%, 0.82% and 2.52%. Head grades of the four bulk samples ranged from 0.24 to 1.10 g Au/t and from 5 to 30 g Ag/t. Sulfide sulfur grades were 0.02 to 0.13%.

Cyanide soluble gold to fire assay ("CN/FA") ratios for the oxide samples generally were greater than 80%. CN/FA ratios for the transitional samples were variable, and tended to decrease with sulfide sulfur head grade, as shown in Figure 13.1. CN/FA was highly variable for the unoxidized samples, and was not correlated to sulfide sulfur content, sample depth or sample elevation.

Figure 13.1 DeLamar Area 2018-2019 Composites: CN/FA vs. Sulfide Sulfur (%)



13.4.1.1 DeLamar Heap Leach Testing

Bottle-roll cyanidation tests were conducted on each of 103 of the drill-core composites and the four bulk samples, at an 80% -1.7mm feed size, to evaluate potential for heap leaching. Summary testing conditions and results from the bottle-roll tests on composites from the DeLamar, DeLamar North, Glen Silver and Sommercamp areas are shown in Table 13.9, and for the Sullivan Gulch composites in Table 13.10. Gold recovery versus sulfide sulfur grade for the oxide, transitional and unoxidized material types is shown in Figure 13.2. Nearly all of the Sullivan Gulch samples tested were unoxidized material, and recoveries from those composites were not correlated to sulfide sulfur grade.

Table 13.9 Summary 2018-2019 DeLamar Bottle-Roll Test Results, 80%-1.7mm Feed Size

Sample Description							Au Recovery, %	Head Grade, gAu/t	Ag Recovery, %	Head Grade, gAg/mt	Reagent Requirements, kg/t ore	
Composite	Type	Oxidation	Sulfide Sulfur, % S	Drill Hole	Interval, meters from	to					NaCN Cons.	Lime Added
DeLamar Area												
4307-B	Bulk Sample	ox	0.19	N/A	N/A		75.0	0.24	40.0	5	0.11	4.2
4307-162	Core	ox	0.25	IDM18_028	23.5	29.7	58.1	0.31	41.7	36	0.22	2.3
4307-A	Bulk Sample	trans	0.48	N/A	N/A		66.4	1.10	53.3	15	0.48	7.4
4307-C	Bulk Sample	trans	0.06	N/A	N/A		81.0	0.42	43.3	30	0.14	3.4
4307-D	Bulk Sample	trans	0.18	N/A	N/A		56.5	0.62	30.0	10	0.17	3.7
4307-161	Core	trans	0.30	IDM18_017	19.5	27.1	83.3	0.18	55.7	70	0.15	2.1
4307-163	Core	mixed (trans/unox)	0.70	IDM18_028	34.1	39.3	27.5	0.40	45.7	35	0.22	2.3
4307-144	Core	unox	0.54	IDM18_025	28.7	33.5	48.4	0.31	36.8	38	0.15	2.0
4307-067	AR	unox	0.62	IDM18_039	55.2	74.7	4.7	0.43	18.9	53	0.07	1.5
DeLamar North Area												
4307-059	AR	mixed (trans/unox)	2.33	IDM18_027	15.2	31.2	13.6	0.44	42.9	49	0.30	2.0
4307-060	AR	mixed (trans/unox)	2.13	IDM18_027	36.0	60.7	20.0	0.40	45.6	57	0.45	2.8
Glen Silver												
4307-048	AR	ox	0.19	IDM18_009	18.3	38.1	72.9	0.48	45.5	11	0.23	5.0
4307-049	AR	ox	0.05	IDM18_009	53.3	62.5	83.7	0.43	35.7	14	0.23	2.0
4307-050	AR	ox	0.11	IDM18_009	64.0	82.3	90.5	0.63	37.5	8	0.15	2.0
4307-055	AR	ox	0.11	IDM18_023	41.5	52.4	80.8	0.52	42.3	26	0.22	2.0
4307-167	Core	ox	0.03	IDM19-131	15.1	22.9	76.9	0.35	30.0	10	0.07	2.6
4307-168	Core	ox	0.07	IDM19-131	39.3	46.0	71.9	0.33	31.8	23	0.15	0.9
4307-169	Core	ox	0.04	IDM19-131	61.0	67.1	77.6	0.64	35.6	69	0.07	1.3
4307-186	Core	ox	0.13	IDM19_117	31.1	40.2	79.7	0.77	14.6	43	0.07	4.2
4307-187	Core	ox	0.05	IDM19_117	61.6	82.9	85.3	0.30	30.0	8	0.00	1.5
4307-051	AR	mixed (ox/trans)	0.38	IDM18_009	89.9	97.5	75.0	0.52	33.3	6	0.15	2.0
4307-188	Core	trans	0.58	IDM19_117	82.9	96.6	62.0	0.45	50.0	6	0.22	1.4
4307-052	AR	mixed (trans/unox)	1.26	IDM18_009	97.5	108.2	25.6	0.43	20.0	5	0.37	5.1
4307-170	Core	mixed (trans/unox)	1.09	IDM19-131	82.6	88.7	33.3	0.77	50.0	10	0.22	2.0
4307-053	AR	unox	2.53	IDM18_009	111.3	121.9	14.0	0.43	16.7	6	0.30	5.9
4307-054	AR	unox	1.91	IDM18_009	146.3	164.6	13.2	0.38	20.0	5	0.30	2.6
4307-056	AR	unox	2.18	IDM18_023	66.4	85.8	7.4	0.68	14.3	7	0.30	5.0
4307-057	AR	unox	2.33	IDM18_023	87.8	116.7	6.6	1.22	10.0	10	0.45	2.7
4307-058	AR	unox	2.11	IDM18_023	116.7	127.6	7.8	1.41	18.2	11	0.45	2.0
Sommercamp												
4307-061	AR	ox	0.08	IDM18_029	19.8	29.0	80.0	0.30	42.1	19	0.15	5.0
4307-180	Core	ox	0.20	IDM19_116	0.0	11.9	80.8	0.31	35.7	15	0.15	2.2
4307-195	Core	ox	0.06	IDM19_134	26.5	35.7	77.8	0.53	32.6	45	0.00	2.6
4307-196	Core	ox	0.06	IDM19_134	54.1	63.1	72.9	0.73	33.3	12	0.07	1.0
4307-197	Core	ox	0.04	IDM19_134	76.8	82.9	67.7	0.32	25.0	8	0.14	1.1
4307-181	Core	mixed (trans/unox)	0.72	IDM19_116	16.8	26.5	5.3	0.22	20.0	4	0.22	3.2
4307-062	AR	unox	0.71	IDM18_029	118.9	131.1	7.0	0.43	14.3	4	0.30	1.5
4307-063	AR	unox	1.07	IDM18_029	135.6	147.8	11.3	1.59	34.1	44	0.30	2.0
4307-064	AR	unox	1.92	IDM18_029	152.4	172.2	17.8	0.45	31.3	16	0.45	2.0
4307-065	AR	unox	3.11	IDM18_029	172.2	189.0	19.3	0.57	25.0	8	0.38	2.0
4307-066	AR	unox	2.76	IDM18_029	196.6	213.4	13.5	0.37	35.7	14	0.08	1.8

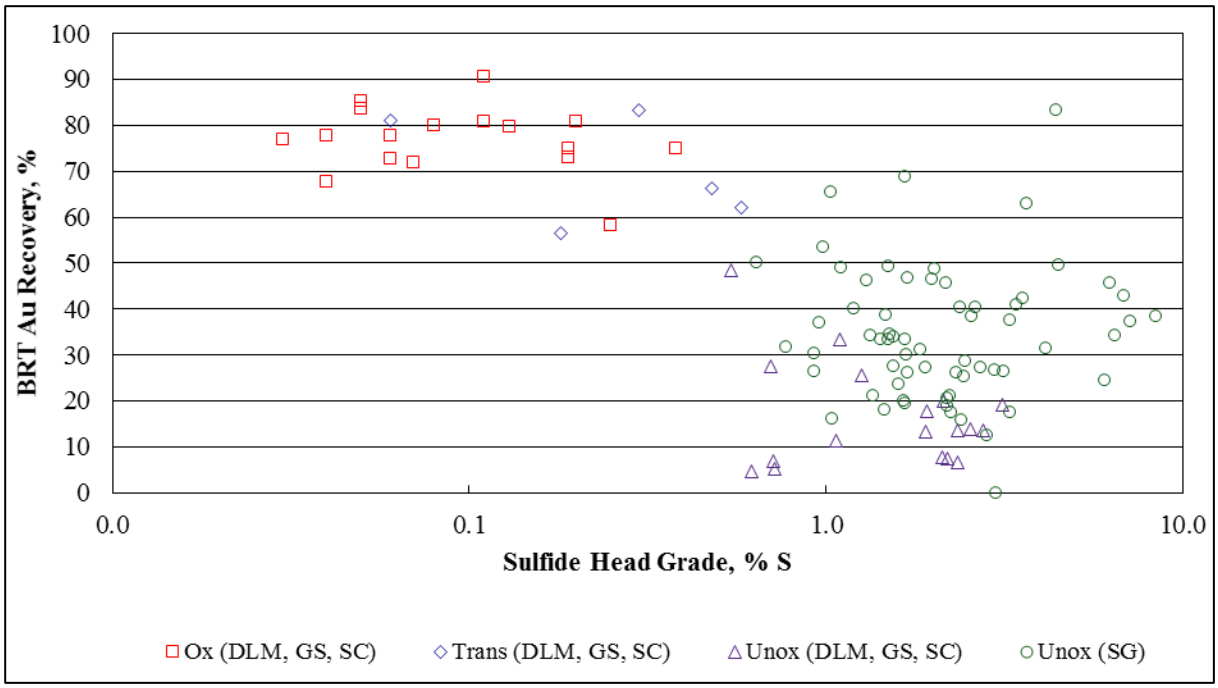
Note: 96 Hour Leach (no interim sampling) at 40% Solids and 1.0 g NaCN/L, DeLamar 2018/2019 Samples; AR denotes assay reject composites. Core denotes split drill core composites. ox denotes oxide; trans denotes transitional; unox denotes unoxidized.

Table 13.10 Summary 2018-2019 Sullivan Gulch Bottle-Roll Tests, 80%-1.7mm Feed Size

Sample Description							Au	Head	Ag	Head	Reagent Requirements, kg/t ore	
			Sulfide	Drill	Interval, meters		Recovery,	Grade,	Recovery,	Grade,	NaCN	Lime
Composite	Type	Oxidation	Grade, % S	Hole	from	to	%	gAu/t	%	gAg/t	Cons.	Added
4307-002	AR	mixed (ox/trans)	0.39	IDM18_005	106.7	121.9	55.3	0.85	30.0	20	0.15	1.0
4307-003	AR	trans	0.19	IDM18_005	121.9	129.5	76.7	0.43	50.0	16	<0.07	1.5
4307-001	AR	mixed (unox/ox)	2.85	IDM18_005	93.0	106.7	47.6	0.42	50.0	10	0.30	2.3
4307-004	AR	unox	1.50	IDM18_005	129.5	152.4	49.3	0.71	28.3	166	0.38	2.5
4307-005	AR	unox	1.04	IDM18_005	152.4	167.6	16.2	1.11	37.8	45	0.15	1.0
4307-006	AR	unox	1.36	IDM18_005	167.6	182.9	21.2	0.66	40.0	50	0.15	1.0
4307-007	AR	unox	1.65	IDM18_005	182.9	198.1	20.0	0.45	38.5	26	0.15	1.1
4307-008	AR	unox	1.67	IDM18_005	198.1	213.4	19.4	0.31	42.1	19	0.15	1.3
4307-009	AR	unox	1.60	IDM18_005	213.4	228.6	23.7	0.38	37.5	24	0.16	1.0
4307-010	AR	unox	3.27	IDM18_005	228.6	243.8	37.5	0.32	43.8	32	0.15	1.5
4307-011	AR	unox	3.57	IDM18_005	248.4	274.3	42.4	0.33	27.3	11	0.08	1.2
4307-012	AR	unox	2.82	IDM18_007	185.9	196.6	12.4	2.01	37.3	59	0.37	2.0
4307-013	AR	unox	2.72	IDM18_007	196.6	213.4	27.3	0.88	26.5	83	0.15	2.7
4307-014	AR	unox	2.32	IDM18_007	213.4	236.2	26.2	0.42	28.9	45	<0.07	4.3
4307-015	AR	unox	1.66	IDM18_007	236.2	259.1	33.3	0.33	50.0	30	0.45	3.9
4307-016	AR	unox	1.99	IDM18_007	259.1	281.9	46.4	0.69	35.6	45	0.30	2.7
4307-017	AR	unox	1.69	IDM18_007	281.9	301.8	26.2	0.42	26.3	57	0.15	1.9
4307-018	AR	unox	1.90	IDM18_007	301.8	320.0	27.3	0.33	37.1	35	0.22	1.7
4307-019	AR	unox	2.45	IDM18_007	320.0	335.3	28.6	0.63	38.5	39	0.30	1.8
4307-020	AR	unox	1.68	IDM18_008	71.6	86.9	30.0	0.30	42.9	7	0.15	2.4
4307-021	AR	unox	0.93	IDM18_008	97.5	112.8	30.4	0.46	48.6	37	0.15	2.5
4307-022	AR	unox	1.47	IDM18_008	112.8	128.0	38.6	0.44	38.1	21	0.30	2.9
4307-023	AR	unox	1.55	IDM18_008	128.0	143.3	27.5	0.40	27.3	326	0.30	3.2
4307-024	AR	unox	3.00	IDM18_008	257.6	265.2	0.0	0.30	33.3	12	0.08	1.0
4307-025	AR	unox	2.01	IDM18_011	71.6	83.8	48.6	3.13	32.1	442	0.38	1.0
4307-026	AR	unox	2.19	IDM18_011	83.8	99.1	20.5	0.44	33.3	51	0.15	1.0
4307-027	AR	unox	2.40	IDM18_011	99.1	114.3	15.8	0.76	27.3	143	0.30	1.0
4307-028	AR	unox	1.84	IDM18_011	114.3	135.6	31.1	0.45	39.5	43	0.30	1.5
4307-029	AR	unox	2.57	IDM18_011	135.6	147.8	38.5	2.34	31.2	452	0.75	3.3
4307-030	AR	unox	2.62	IDM18_011	153.9	179.8	40.5	0.37	39.3	28	0.15	1.8
4307-031	AR	unox	2.22	IDM18_011	185.9	222.5	21.1	0.38	40.6	32	0.15	1.0
4307-032	AR	unox	3.29	IDM18_011	233.2	266.7	17.4	0.46	35.1	37	0.23	1.0
4307-033	AR	unox	2.38	IDM18_012	301.8	323.1	40.3	0.67	32.4	37	0.15	1.0
4307-034	AR	unox	1.67	IDM18_012	336.8	359.7	68.9	0.61	50.0	8	0.15	1.0
4307-035	AR	unox	2.18	IDM18_014	166.1	182.9	18.8	0.32	44.4	54	0.30	2.9
4307-036	AR	unox	2.43	IDM18_014	182.9	204.2	25.3	0.95	31.3	64	0.38	5.0
4307-037	AR	unox	4.41	IDM18_014	204.2	210.3	83.4	6.32	44.7	226	0.37	8.1
4307-038	AR	unox	4.48	IDM18_014	210.3	228.6	49.7	1.57	40.7	150	0.23	7.4
4307-039	AR	unox	6.23	IDM18_014	228.6	243.8	45.7	2.47	37.0	73	0.67	5.4
4307-040	AR	unox	7.12	IDM18_014	243.8	259.1	37.4	2.06	34.2	202	0.07	3.3
4307-041	AR	unox	3.66	IDM18_014	259.1	272.8	62.9	2.64	45.5	200	0.45	2.3
4307-042	AR	unox	6.84	IDM18_014	274.3	289.6	42.9	0.83	50.0	65	0.37	1.5
4307-043	AR	unox	4.13	IDM18_014	289.6	320.0	31.5	0.63	33.3	28	<0.07	1.3
4307-044	AR	unox	6.44	IDM18_014	320.0	350.5	34.3	0.81	38.9	37	<0.07	1.5
4307-045	AR	unox	6.05	IDM18_014	350.5	371.9	24.6	0.57	34.8	21	0.22	0.6
4307-046	AR	unox	8.44	IDM18_014	371.9	396.2	38.5	1.05	37.3	52	0.75	0.8
4307-071	AR	unox	1.03	IDM18-046	275.8	285.0	65.6	12.63	37.0	53	0.08	1.0
4307-072	AR	unox	0.98	IDM18-046	285.0	295.7	53.5	1.28	22.2	7	0.07	1.0
4307-073	AR	unox	1.10	IDM18-046	295.7	306.3	49.0	1.11	20.0	5	0.00	1.0
4307-074	AR	unox	1.50	IDM18-046	312.4	324.6	33.3	1.18	14.3	6	0.15	1.0
4307-075	AR	unox	0.64	IDM18-046	338.3	349.0	50.0	0.52	18.2	7	0.07	1.0
4307-076	AR	unox	0.96	IDM18-046	353.6	373.4	37.0	0.56	16.7	3	0.15	1.0
4307-077	AR	unox	1.30	IDM18-046	373.4	381.0	46.2	0.36	12.5	4	0.15	1.0
4307-080	AR	unox	1.55	IDM18-047	263.7	275.8	34.0	1.36	35.5	140	0.30	2.8
4307-081	AR	unox	1.20	IDM18-047	275.8	283.5	40.0	2.42	25.9	165	0.22	1.2
4307-082	AR	unox	1.69	IDM18-047	359.7	379.5	46.9	1.16	40.7	123	0.22	1.5
4307-085	AR	unox	0.77	IDM18-048	115.8	129.5	31.8	0.70	64.7	18	0.08	3.0
4307-086	AR	unox	0.93	IDM18-048	132.6	143.3	26.3	0.60	71.9	30	0.15	3.0
4307-087	AR	unox	1.51	IDM18-048	144.8	160.0	34.6	0.77	50.0	29	0.22	3.0
4307-088	AR	unox	1.42	IDM18-048	192.0	202.7	33.3	0.74	43.8	21	0.23	2.9
4307-089	AR	unox	1.33	IDM18-048	228.6	249.9	34.2	0.96	34.8	21	0.22	1.8
4307-090	AR	unox	2.96	IDM18-048	266.7	278.9	26.8	0.42	37.5	17	0.15	1.6
4307-091	AR	unox	3.42	IDM18-048	280.4	295.7	40.8	0.58	38.5	27	0.15	2.0
4307-092	AR	unox	1.46	IDM18-048	320.0	333.8	18.2	0.41	38.7	32	<0.07	2.0
4307-093	AR	unox	2.17	IDM18-048	336.8	344.4	45.6	0.94	36.4	18	0.15	2.0
4307-094	AR	unox	3.16	IDM18-048	373.4	387.1	26.5	0.64	29.8	35	0.30	2.0
4307-095	AR	unox	2.24	IDM18-048	411.5	428.2	17.4	0.45	20.0	13	<0.07	2.0

Note: 96 Hour Leach Time (no interim sampling), 40% Solids, 1.0 g NaCN/L, Sullivan Gulch 2018/2019 Composites; AR denotes assay reject composites. Core denotes split drill core composites. Ox denotes oxide; trans denotes transitional; unox denotes unoxidized.

Figure 13.2 DeLamar 2018-2019 Composites Bottle-Roll Tests, Gold Recovery vs. Sulfide Sulfur
(1.7mm)



Bottle-roll tests on nine Glen Silver and five Sommercamp oxide composites indicate good potential for heap leaching. Gold recoveries obtained from those composites in 96 hours of leaching ranged from 67.7% to 90.5% and averaged 78.5%. Corresponding silver recoveries averaged 33.7%. No transitional material composites were available from Sommercamp. A single transitional material composite from Glen Silver gave a gold and silver recovery of 62.0% and 50.0%, respectively.

In the case of the DeLamar/DeLamar North area, the number of oxide samples was more limited. The DeLamar area bulk oxide sample (4307-B) gave a 75.0% gold recovery and 40.0% silver recovery. A single oxide drill core composite from this area was tested and gave gold and silver recoveries of 58.1% and 41.7%, respectively. The DeLamar area transitional type bulk samples gave gold recoveries that ranged from 56.5% to 81.0%. Transitional core composite 4307-161 gave an 83.3% gold recovery.

Cyanide consumption for the 1.7mm bottle-roll tests on oxide and transitional-material type samples was low and averaged 0.15 kg NaCN/t. Lime demand was more variable and averaged 2.7 kg/t.

The unoxidized and mixed transitional/unoxidized composites tested generally gave very low gold recoveries (16.5% average), indicating poor potential for heap leaching. Those composites had an average sulfide sulfur grade of 1.67%.

Most of the Sullivan Gulch composites that were bottle-roll tested were classified as unoxidized material type samples. Gold recoveries from the unoxidized composites generally were poor (34.6% average), indicating poor potential for heap leaching. This is not unexpected, considering the generally higher sulfide sulfur content of this material. As discussed later in this section, ongoing metallurgical testing

indicates the DeLamar unoxidized material type responds significantly better to upgrading by gravity and flotation methods. A single transitional composite and a mixed (ox/trans) composite tested gave an average gold recovery of 66.0%, by bottle-roll cyanidation at the -1.7mm size.

A column-leach test was completed on each of the four DeLamar bulk samples at an 80% -13mm (0.5 inch) feed size. Results from those tests are shown with comparative bottle-roll test results in Table 13.11 and in Figure 13.3 and Figure 13.4.

Table 13.11 DeLamar 2018-2019 Summary Column-Leach Cyanidation Tests on Bulk Samples

Feed Size	Test Type	Leach Time, Days	Au Rec., %	Calc'd. Head, gAu/t	Ag Rec. %	Calc'd. Head, gAg/t	Reagents, kg/mt ore		
							NaCN Consumed	Lime Added	Cement Added
Bulk Sample 4307-A (Trans Clay)									
80%-13mm	CLT	79	73.4	1.09	50.0	16	1.27	N/A	10.0
80%-1.7mm	BRT	4	66.4	1.10	53.3	15	0.48	7.4	N/A
Bulk Sample 4307-B (Oxide)									
80%-13mm	CLT	68	87.5	0.24	25.0	4	0.45	N/A	7.5
80%-1.7mm	BRT	4	75.0	0.24	40.0	5	0.11	4.2	N/A
Bulk Sample 4307-C (Trans Hard)									
80%-13mm	CLT	77	92.5	0.40	20.0	35	0.66	N/A	5.0
80%-1.7mm	BRT	4	81.0	0.42	43.3	30	0.14	3.4	N/A
Bulk Sample 4307-D (Trans Hard)									
80%-13mm	CLT	79	67.7	0.65	19.5	10	1.17	N/A	5.0
80%-1.7mm	BRT	4	56.5	0.62	30.0	10	0.17	3.7	N/A

Note: Column tests were conducted in 6 inch diameter x 10 foot high columns, using a solution application rate of 9.6 liters per hour per square meter and a cyanide concentration of 1.0 gNaCN/L.

Figure 13.3 Gold Leach Rate Profiles for Column-Leach Tests, DeLamar Bulk Samples,

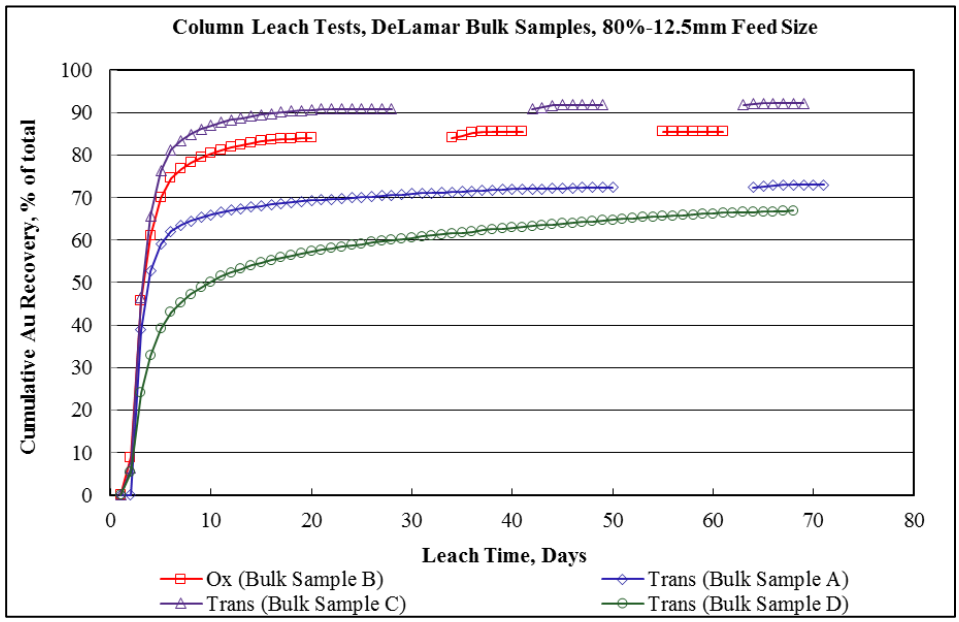
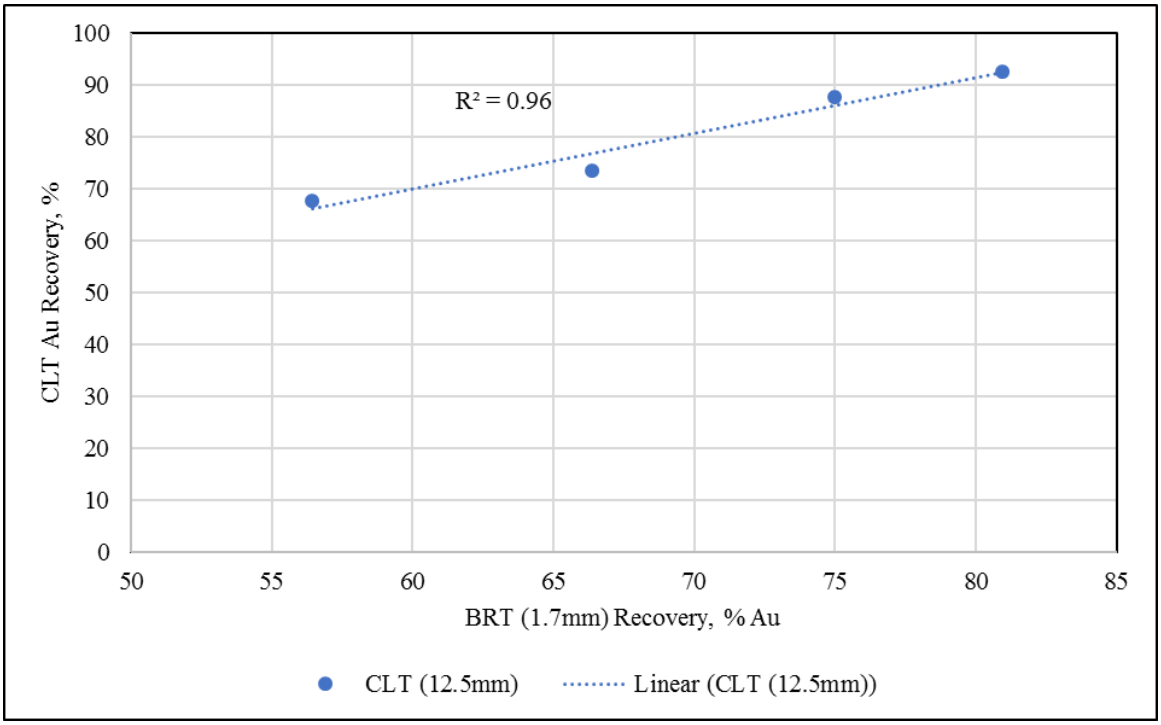


Figure 13.4 Gold Recovery in Column-Leach Tests vs. Bottle Roll Tests, DeLamar Bulk Samples



Column-test results show that the DeLamar oxide and transitional samples were amenable to heap-leach cyanidation treatment, at an 80% -12.5mm feed size. Gold and silver recoveries obtained from the oxide sample were 87.5% and 25.0%, respectively. Average gold and silver recoveries obtained from the three transitional samples were 77.9% and 29.8%, respectively.

The column charges were agglomerated using cement before leaching. Head screen analysis results showed that the samples did not have excessive fines content (all contained <13% by weight material passing 75µm in size) and may not have required agglomeration pre-treatment. These data demonstrate that some of the DeLamar heap-leach feed will likely not require agglomeration pretreatment. Optimization of the heap-leach flowsheet will continue with further studies. Cyanide consumptions were moderate (0.45 – 1.27 kg NaCN/mt).

Column-leach testing was in progress on the same bulk samples at 80% -50mm and 100% -200mm feed sizes as of the effective date of this report. Results from those tests were not available, but available data are encouraging and indicate that consideration of two-stage crushing for the DeLamar heap-leach feed will be required.

Bottle-roll test gold recoveries (1.7mm feed size) and column-leach test gold recoveries (13mm feed size) were strongly correlated. That correlation is shown in Figure 13.4. The observed correlation can be used to predict column-leach test gold recoveries from available drill core bottle-roll test results. That correlation was used to predict average column-test gold recoveries of 87% for the oxide drill core composites and 85% for the transitional drill core composites (82% if the bulk samples are included).

13.4.1.2 DeLamar Agitated Cyanide Leach Testing

Thirty unoxidized composites from DeLamar, Glen Silver, Sommercamp and Sullivan Gulch, along with three transitional, one oxide and two mixed (trans/unox) composites, were used for a bottle-roll leach test at an 80% -75µm (200 mesh) feed size, to evaluate amenability to "whole-ore" milling/cyanidation treatment. Summary testing conditions and results from those tests are presented in Table 13.12.

Gold recoveries obtained from the one oxide and three transitional composites ranged from 61.8% to 88.9%, in 96 hours of leaching, indicating good potential for milling/cyanidation treatment.

Gold recoveries obtained from the unoxidized and mixed (trans/unox) composites, at the 75µm feed size, were highly variable, ranging from 6.8% to 81.1%. Gold recovery was not correlated to sulfide sulfur content, sample depth or elevation. Silver recoveries also were variable and ranged from 14.3% to 76.7%. Recovery distribution data are shown for gold and silver recoveries in Figure 13.5 and Figure 13.6, respectively. While these results indicate the potential for milling/cyanidation treatment for a relatively small portion of the DeLamar unoxidized material, particularly from the Sullivan Gulch area, the factors controlling recovery are yet poorly understood. Further testing and mineralogical characterization are planned to obtain a better understanding of this variability, in order to determine what portion of the unoxidized resource may be economically processed by grind-leach.

Table 13.12 Grind Agitated Bottle Roll Tests, DeLamar 2018-2019 Drill-Core Composites

Sample Description							Au Recovery, %	Head Grade, gAu/t	Ag Recovery, %	Head Grade, gAg/t	Reagent Requirements, kg/t ore	
Composite	Zone	Type	Oxidation	Drill Hole	Interval, meters from	to					NaCN Cons.	Lime Added
4307-162	DeLamar North	Core	ox	IDM18_028	23.3	29.7	61.8	0.34	80.0	35	1.54	2.8
4307-144	DeLamar	Core	trans	IDM18_025	28.6	33.5	64.3	0.42	61.1	36	0.62	3.4
4307-161	DeLamar North	Core	trans	IDM18_017	19.5	27.1	88.9	0.18	84.4	77	1.00	2.7
4307-163	DeLamar North	Core	mixed[1)]	IDM18_028	34.1	39.3	34.3	0.35	76.5	34	1.78	2.5
4307-164	DeLamar North	Core	unox	IDM18_028	48.5	57.6	59.4	0.64	76.7	30	1.56	4.2
4307-181	Sommercamp	Core	mixed[1)]	IDM19_116	16.8	26.5	33.3	0.27	50.0	4	0.52	3.6
4307-182	Sommercamp	Core	unox	IDM19_116	151.5	160.6	28.6	0.28	38.1	21	0.27	2.1
4307-183	Sommercamp	Core	unox	IDM19_116	168.2	174.3	43.8	0.32	46.9	32	0.41	1.9
4307-184	Sommercamp	Core	unox	IDM19_116	191.1	198.7	38.5	0.26	44.4	45	0.26	1.9
4307-185	Sommercamp	Core	unox	IDM19_116	220.8	229.2	19.4	0.72	37.5	8	0.77	2.6
4307-198	Sommercamp	Core	unox	IDM19_134	126.5	137.5	11.1	0.63	44.4	9	0.71	3.6
4307-119	Glen Silver	AR	unox	IDM18_023	87.6	127.6	12.5	1.20	22.2	9	0.90	4.3
4307-145/146MC	Glen Silver	Core	unox	IDM18_013/IDM18_030	54.6	72.1	33.3	0.75	50.0	12	0.64	3.0
4307-147/148MC	Glen Silver	Core	unox	IDM18_030	97.2	155.9	15.1	0.53	44.4	9	0.17	2.2
4307-188	Glen Silver	Core	trans	IDM19_117	82.9	96.6	67.8	0.59	75.0	4	0.23	2.2
4307-189	Glen Silver	Core	unox	IDM19_117	96.6	114.9	13.0	0.46	66.7	12	0.24	2.2
4307-190	Glen Silver	Core	unox	IDM19_117	146.9	162.2	10.9	1.01	25.0	8	0.52	3.9
4307-191	Glen Silver	Core	unox	IDM19_117	176.5	186.2	7.3	1.10	40.0	10	0.59	2.9
4307-192	Glen Silver	Core	unox	IDM19_117	186.2	190.0	70.6	0.68	60.0	10	0.72	8.9
4307-193	Glen Silver	Core	unox	IDM19_133	34.1	40.2	16.9	0.71	14.3	7	0.43	3.7
4307-194	Glen Silver	Core	unox	IDM19_133	52.1	63.1	20.5	0.73	42.9	14	0.57	3.0
4307-171	Glen Silver	Core	unox	IDM19_131	94.8	100.9	14.6	1.30	38.1	11	5.44	3.0
4307-172	Glen Silver	Core	unox	IDM19_131	122.2	128.3	6.8	0.73	31.3	8	0.27	3.0
4307-178	Glen Silver	Core	unox	IDM19_131	107.0	113.1	20.0	0.35	42.3	5	1.04	4.6
4307-005	Sullivan Gulch	AR	unox	IDM18_005	152.4	167.6	21.3	1.08	52.4	42	0.84	2.3
4307-012	Sullivan Gulch	AR	unox	IDM18_007	185.9	196.6	19.3	1.81	54.5	66	0.80	4.5
4307-025	Sullivan Gulch	AR	unox	IDM18_011	71.6	83.8	59.9	3.09	32.7	505	0.87	2.1
4307-029	Sullivan Gulch	AR	unox	IDM18_011	135.6	147.8	45.0	2.29	30.6	480	1.49	5.4
4307-046	Sullivan Gulch	AR	unox	IDM18_014	371.9	396.2	67.3	1.10	47.4	57	0.70	2.6
4307-047	Sullivan Gulch	AR	unox	IDM18_014	204.2	272.8	70.7	2.46	56.2	153	1.86	5.7
4307-120	Sullivan Gulch	AR	unox	IDM18_046	275.8	324.6	81.1	5.19	50.0	24	0.17	2.0
4307-121	Sullivan Gulch	AR	unox	IDM18_047	263.7	283.5	63.2	1.71	44.4	153	0.86	2.7
4307-149-153MC	Sullivan Gulch	Core	unox	IDM18_052	207.0	342.7	45.3	1.06	41.3	46	0.24	3.0
4307-154	Sullivan Gulch	Core	unox	IDM18_055	190.0	204.2	47.8	0.69	28.6	28	0.17	2.0
4307-165	Sullivan Gulch North	Core	unox	IDM18_053	109.4	118.6	20.7	0.29	40.0	15	1.50	5.1
4307-166	Sullivan Gulch North	Core	unox	IDM18_054	80.3	88.1	40.0	0.35	29.3	41	0.23	1.2

Note: 80% -75µm Feed Size, 72 Hour Leach Time (with interim sampling and reagent maintenance), 40% Solids, 1.0 g NaCN/L; R denotes assay reject composites. Core denotes split drill core composites; [1)] Mixed composite containing transitional and unoxidized material types.

Figure 13.5 Gold Recovery, Unoxidized Sample Grind-Leach, 2018-2019 Drill Core Composites
(80%-75µm Feed Size)

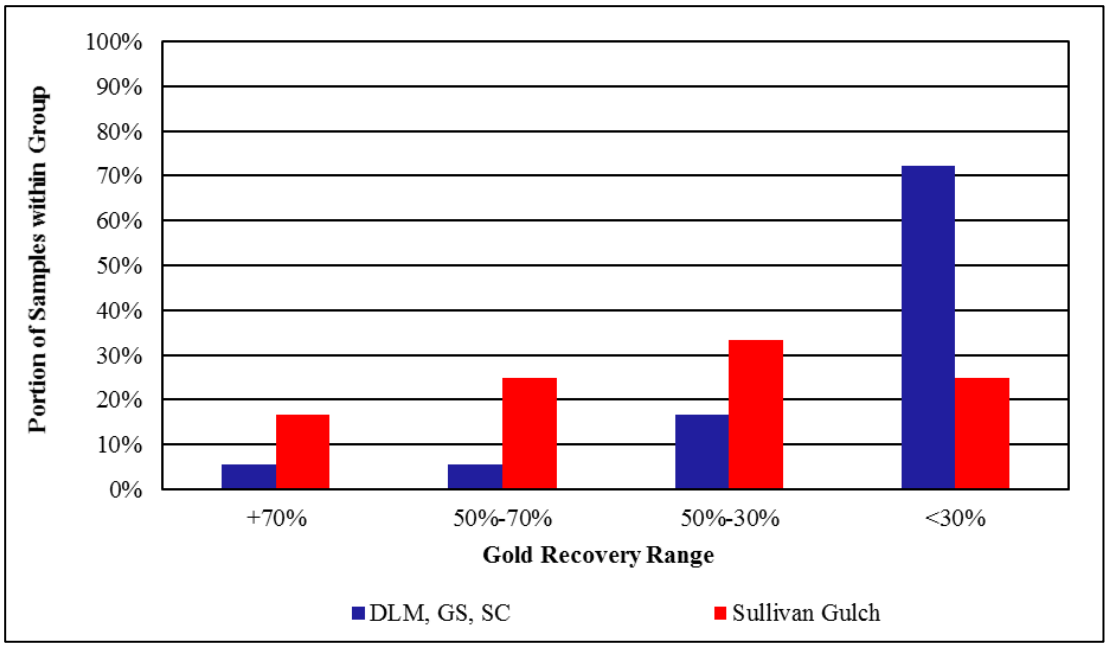
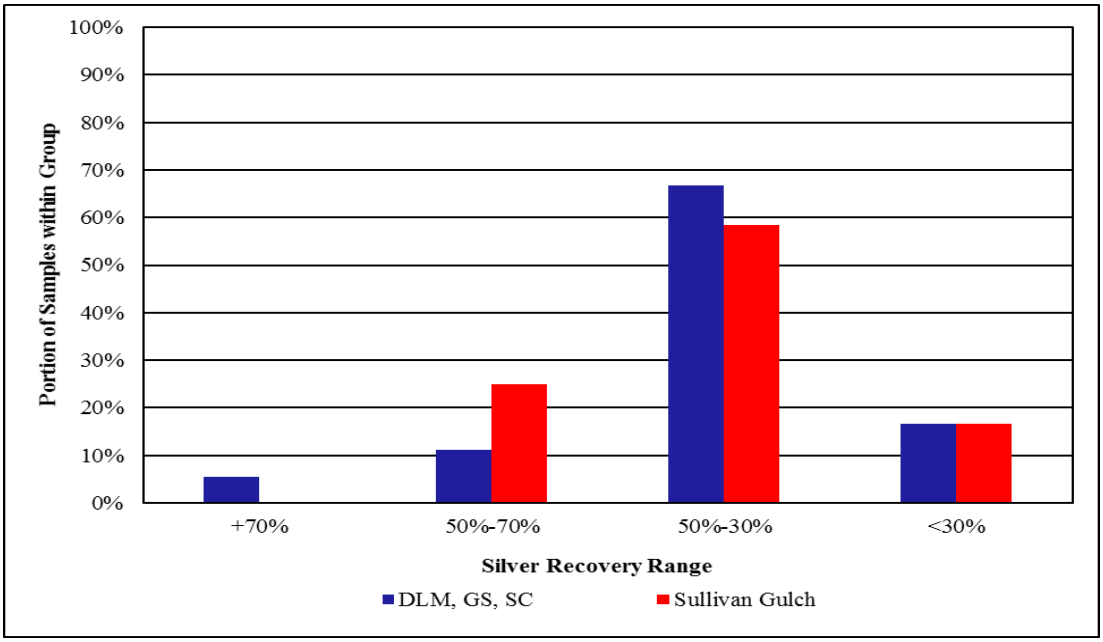


Figure 13.6 Silver Recovery, Unoxidized Sample Grind-Leach, 2018-2019 Drill Core Composites
(80%-75µm Feed Size)



13.4.1.3 DeLamar Gravity Concentration and Flotation Testing

A series of scoping-level gravity concentration tests, with bulk sulfide flotation on the gravity tailings, was conducted on nine unoxidized drill core composites from Sullivan Gulch and Glen Silver. These tests were conducted to obtain preliminary information regarding the effectiveness of upgrading the DeLamar unoxidized material by conventional gravity concentration and flotation processing methods. The samples tested included eight composites from Sullivan Gulch and one composite from Glen Silver. Samples of 1.0 kg were ball milled to 80% -75µm, subjected to gravity concentration and the resulting gravity rougher tailings were subjected to bulk sulfide flotation treatment. Summary results from those tests are shown in Table 13.13.

The Sullivan Gulch composites generally responded well to gravity concentration, followed by bulk sulfide flotation treatment, at an 80% -75µm feed size. The gravity rougher concentrates contained between 2.7% and 5.6% of the "whole-ore" mass and represented average gold and silver recoveries of 34.9% and 26.4%, respectively. The resulting gravity tailings generally responded well to bulk sulfide flotation treatment. The combined gravity/flotation rougher concentrate produced from six of the eight Sullivan Gulch composites tested was equivalent to an average of 19% of the "whole-ore" weight and contained an average of 89% of the total gold and 91% of the total silver. The remaining two Sullivan Gulch composites also gave high gold and silver recoveries, but the mass pull during flotation was anomalously high (about 40% of the "whole-ore" weight). Carry-over of clay minerals to the concentrates appeared to be responsible for the higher mass pulls observed during these tests. It is expected that through optimization of flotation conditions, response of the composites to flotation treatment will be improved.

Based on results from the preliminary tests described above, two master composites each were prepared from unoxidized Sullivan Gulch and Glen Silver materials. Those composites were used for optimization of flotation conditions. Flotation testing on those composites was conducted without gravity concentration. Parameters evaluated included feed size for all four composites, along with reagent additions and rougher concentrate regrind for the two Sullivan Gulch composites. A total of 24 tests were conducted on the four composites. A summary of representative test results is shown in Table 13.14.

Both Sullivan Gulch composites responded reasonably well to bulk sulfide flotation treatment, with gold and silver recoveries to the flotation rougher concentrates of approximately 90% to 95%. Selectivity for the initial tests was lower than desired, and flotation rougher concentrate mass pulls were generally about 20% to 30%.

Optimization testing was successful in decreasing flotation mass pull to about 9% to 11%, while maintaining gold recovery (either to cleaner or rougher concentrate) at between 86% to over 90%, and silver recovery above 90%. Optimization testing is ongoing, and it is expected that gold and silver recoveries in excess of 90% will be obtainable at a concentrate mass pull of about 10% to 13%. Grind optimization test results indicate that a grind size of as coarse as 80% -150µm will likely be sufficient for maximizing gold and silver recoveries by flotation of the Sullivan Gulch unoxidized material. Rougher flotation recoveries appeared to be incrementally lower at a coarser (80% -212µm) feed size.

Table 13.13 Gravity Concentration with Flotation of Gravity Rougher Tailings

(DeLamar 2018-2019 Composites, 80%-75µm Feed Size)

Area:	Sullivan Gulch								Glen Silver
Composite:	4307-005	4307-012	4307-025	4307-029	4307-046	4307-047	4307-120	4307-121	4307-119
Gravity Test No.:	G-1	G-2	G-3	G-4	G-5	G-6	G-8	G-9	G-7
Flotation Test No.:	F-7	F-8	F-9	F-10	F-11	F-12	F-14	F-15	F-13
Sulfide Grade, % S	1.04	2.82	2.01	2.57	8.44	5.28	1.30	1.41	2.27
Weight, %									
Gravity Cl. Conc.	0.7	1.3	1.2	1.0	2.1	0.6	1.1	1.4	1.2
Gravity Cl. Tail	2.2	3.2	3.1	2.7	3.5	2.8	1.6	2.0	2.5
Flotation Cl. Conc.	7.9	4.3	6.0	10.2	11.5	12.4	3.3	4.7	4.0
Combined (Grav.+ Flot. Cl. Conc.)	10.1	7.5	9.1	12.9	15.0	15.2	4.9	6.7	6.5
Flotation Cl. Tail	15.3	14.2	33.7	9.7	29.9	10.0	5.5	6.7	19.4
Combined (Grav.+ Flot. Ro. Conc.)	25.4	21.7	42.8	22.6	44.9	25.2	10.4	13.4	25.9
Flotation Ro. Tail	73.9	77.0	56.0	76.4	53.0	74.2	88.5	85.3	72.9
Grade, g Au/t									
Gravity Cl. Conc.	18.2	23.9	95.4	32.4	19.5	94.8	185.0	54.2	16.0
Gravity Cl. Tail	2.43	5.68	3.87	3.39	5.28	11.40	19.50	3.37	4.50
Flotation Cl. Conc.	6.07	10.80	9.77	11.60	3.37	10.70	46.40	9.38	6.91
Combined (Grav.+ Flot. Cl. Conc.)	6.54	12.76	20.34	12.39	6.55	14.57	79.15	18.91	8.94
Flotation Cl. Tail	1.68	2.83	2.02	3.64	0.54	1.49	19.30	1.37	1.97
Combined (Grav.+ Flot. Ro. Conc.)	3.61	6.26	5.92	8.64	2.55	9.38	47.50	10.14	3.72
Flotation Ro. Tail	0.20	0.57	0.72	0.29	0.15	0.11	0.17	0.23	0.46
Calculated Head	1.07	1.80	2.93	2.17	1.22	2.45	5.09	1.56	1.30
Grade, g Ag/t									
Gravity Cl. Conc.	610	730	9,470	5,950	447	2,810	483	5,820	104
Gravity Cl. Tail	107	174	723	444	266	534	77	393	31
Flotation Cl. Conc.	229	420	2,610	3,230	172	796	320	1,530	61
Combined (Grav.+ Flot. Cl. Conc.)	245	442	3,216	3,108	257	859	349	2,407	69
Flotation Cl. Tail	60	86	426	541	40	120	104	143	17
Combined (Grav.+ Flot. Ro. Conc.)	133	209	1,019	2,006	112	566	219	1,275	30
Flotation Ro. Tail	9	16	100	14	9	10	1	12	3
Calculated Head	41	58	492	464	55	150	24	181	10
Au Distribution, % of total									
Gravity Cl. Conc.	12.0	17.3	39.0	14.9	33.5	23.3	40.0	48.8	14.8
Gravity Cl. Tail	5.0	10.1	4.1	4.2	15.1	13.1	6.1	4.3	8.7
Flotation Cl. Conc.	45.0	25.8	20.0	54.4	31.7	54.2	30.1	28.4	21.3
Combined (Grav.+ Flot. Cl. Conc.)	62.0	53.2	63.1	73.5	80.3	90.6	76.2	81.5	44.8
Flotation Cl. Tail	24.1	22.4	23.2	16.3	13.2	6.1	20.9	5.9	29.4
Combined (Grav.+ Flot. Ro. Conc.)	86.1	75.6	86.3	89.8	93.5	96.7	97.1	87.4	74.2
Flotation Ro. Tail	13.9	24.4	13.7	10.2	6.5	3.3	3.0	12.6	25.8
Ag Distribution, % of total									
Gravity Cl. Conc.	10.5	16.5	23.1	12.8	17.0	11.2	22.4	45.0	12.5
Gravity Cl. Tail	5.8	9.7	4.6	2.6	16.9	10.0	5.2	4.3	7.8
Flotation Cl. Conc.	44.6	31.3	31.8	71.0	35.8	65.8	44.5	39.7	24.5
Combined (Grav.+ Flot. Cl. Conc.)	60.9	57.5	59.5	86.4	69.7	87.0	72.1	89.0	44.8
Flotation Cl. Tail	22.7	21.2	29.1	11.3	21.7	8.0	24.1	5.3	33.2
Combined (Grav.+ Flot. Ro. Conc.)	83.6	78.7	88.6	97.7	91.4	95.0	96.2	94.3	78.0
Flotation Ro. Tail	16.4	21.3	11.4	2.3	8.6	5.0	3.7	5.7	22.0

Table 13.14 Summary of Flotation Test Results and Optimization Testing, 2018-2019

	Flotation Cleaner Concentrate					Flotation Rougher Concentrate					Head Grade		
	Mass	Grade		Recovery		Mass	Grade		Recovery				
Feed Size	%	g Au/t	g Ag/t	% Au	% Ag	%	g Au/t	g Ag/t	% Au	% Ag	g Au/t	g Ag/t	% S$^{=}$
4307-149-153MC; Sullivan Gulch; IDM18_052; 679' - 1,124'; unox; Tql lithology; mixed alteration													
80%-212µm	9.8	8.8	377	82.2	80.1	15.5	6.20	287	91.9	96.3	1.05	46	2.17
80%-150µm	4.1	18.2	826	73.4	82.5	12.6	7.51	312	93.1	95.7	1.02	41	2.21
80%-75µm	7.7	9.8	545	69.4	90.2	17.8	5.70	252	93.2	96.5	1.09	47	2.28
4307-154; Sullivan Gulch; IDM18_055; 623.5' - 670'; unox; Tpl lithology; mixed alteration													
80%-212µm	7.7	10.7	249	80.8	86.4	19.5	4.69	106	89.7	92.8	1.02	22	3.76
80%-150µm	8.2	8.2	255	87.3	86.0	21.3	3.42	110	94.8	96.8	0.77	24	3.85
80%-75µm	11.4	5.0	206	85.6	92.5	25.9	2.42	95	94.4	97.1	0.66	25	3.31
4307-145/146MC; Glen Silver; IDM18_013/IDM18_030; 179' - 236.65'; unox; Tpr lithology; mixed alteration													
80%-75µm	5.0	7.2	128	48.6	54.1	19.7	2.77	44	73.9	72.8	0.74	12	1.26
80%-45µm	4.4	10.2	151	57.9	60.0	9.6	5.82	87	72.0	75.5	0.78	11	1.32
4307-147/148MC; Glen Silver; IDM18_030; 319' - 511.4'; unox; Tql lithology; mixed alteration													
80%-75µm	6.1	5.1	99	60.7	79.5	13.8	2.83	49	76.4	88.6	0.51	8	1.33
80%-45µm	3.6	6.1	148	45.7	66.5	8.8	3.74	81	67.9	88.6	0.48	8	1.23

Testing on the Glen Silver unoxidized master composites showed that they gave lower flotation recoveries than the Sullivan Gulch material. In general, gold and silver recoveries of about 75% were achieved with flotation rougher concentrate mass pulls of approximately 14% to 19%. Very fine grinding (80% -45µm) was evaluated to determine if rougher flotation recoveries could be improved. Those results indicated that finer grinding was not effective for improving recoveries.

Modified diagnostic leach tests were conducted on the flotation rougher tailings generated from the Glen Silver composites, at the -45µm feed size, to determine gold deportment of values of reporting to the flotation tailings. Results indicate the potential for significantly improving flotation recoveries through continued optimization of flotation conditions, and they suggest the recovery of cyanide-soluble gold from the flotation tailings by leaching can be considered. Mineralogical examination of select flotation tailings is planned to better evaluate causes for gold and silver losses to the flotation rougher tailings from the Glen Silver unoxidized material.

13.4.2 Florida Mountain Area Testing

A summary of the 2018-2019 Florida Mountain metallurgical samples tested at McClelland as of the effective date of this report is shown in Table 13.15.

Table 13.15 Florida Mountain 2018-2019 Drill Hole Composite Summary

Area	Drill Hole	Interval Range, meters [1]		Composites			
		from	to	Oxide	Transitional	Un-Oxidized	Mixed[2]
Florida Mountain	IFM18_001	5.6	16.9	0	1	0	0
Florida Mountain	IFM_18_001A	11.7	313.3	0	7	2	1
Florida Mountain	IFM18_003	0.0	161.8	2	2	6	1
Florida Mountain	IFM_18_003[3]	122.5	169.5	0	0	1	0
Florida Mountain	IFM18_004	122.5	191.4	0	0	5	0
Florida Mountain	IFM18_010	17.7	169.5	0	7	4	0
Florida Mountain	IFM18_012	4.9	119.8	0	2	1	0
Florida Mountain	IFM18_025	10.1	39.0	0	3	0	0
Florida Mountain	IFM18_026A	14.5	111.9	0	2	4	0
Total				**2**	**24**	**23**	**2**

1) *Not all core within range was used for composites. Samples were composited based on oxidation, lithology, alteration, grade and continuity.*
2) *Composite contains more material from multiple oxidation classes.*
3) *Contains drill core from holes IFM18_003, IFM18_004 and IFM18_010.*

A total of 45 drill-hole composites were prepared from seven Florida Mountain drill holes for metallurgical testing. Samples were composited according to oxidation classification, lithology and sample grade. The composites included two oxidized composites, 23 transitional composites and 12 unoxidized composites, prepared from 517.55 meters of drill hole samples.

Gold head assays showed that the Florida Mountain composites ranged in grade from 0.24 to 2.23 g Au/t. Corresponding silver head assays ranged from 1.7 to 342.8 g Ag/t. Sulfide sulfur content of the two oxide composites was 0.01% and 0.04% sulfide sulfur, respectively. Sulfide sulfur content of the transitional composites ranged from <0.01% to 0.36% sulfide sulfur. Sulfide sulfur content of the unoxidized composites ranged from 0.02% to 2.17% sulfide sulfur.

13.4.2.1 Florida Mountain Comminution Testing

Bond (ball mill) work index (grindability) tests were conducted on two select, unoxidized composites (4307-135 and 4307-142). Results showed the material was of medium hardness, with work indices of 14.1 kW-hr/st and 14.9 kW-hr/st, respectively.

13.4.2.2 Florida Mountain Heap Leach Testing

Bottle-roll cyanidation tests were conducted on each of 37 of the 2018-2019 Florida Mountain composites, at an 80% -1.7mm feed size, to evaluate potential for heap leaching. Summary test conditions and results from the bottle-roll tests are shown in Table 13.16.

Table 13.16 Florida Mountain 2018-2019 Bottle-Roll Test Results

(80%-1.7mm Feed Size, 96 Hour Leach Time at 40% Solids and 1.0 g NaCN/L)

Sample Description						Au Recovery, %	Head Grade, g Au/t	Ag Recovery, %	Head Grade, g Ag/t	Reagent Requirements, kg/t ore	
Drill Composite	Type	Oxidation	Hole	Interval, meters from	Interval, meters to					NaCN Cons.	Lime Added
4307-126	Core	ox	IFM18_003	0.0	32.9	83.6	0.73	37.5	8	0.16	1.0
4307-100	AR	ox	IFM18_003	4.0	32.9	74.5	0.47	66.7	6	0.08	1.0
4307-096	AR	trans	IFM18_001	5.6	16.9	82.7	1.27	44.0	25	0.15	1.4
4307-122	Core	trans	IFM18_001A	11.7	20.3	89.4	0.47	62.5	16	0.24	1.1
4307-097	AR	trans	IFM18_001A	39.0	56.4	69.1	0.68	53.7	67	0.23	1.3
4307-123	Core	trans	IFM18_001A	39.0	56.4	83.9	0.31	58.8	80	0.21	1.3
4307-098	AR	trans	IFM18_001A	66.4	87.5	40.6	0.96	33.3	3	0.07	1.0
4307-124	Core	trans	IFM18_001A	66.4	81.7	60.0	0.40	25.0	4	0.08	0.8
4307-101	AR	trans	IFM18_003	43.6	71.2	85.7	0.70	35.3	17	0.15	1.2
4307-127	Core	trans	IFM18_003	43.6	71.2	89.7	0.78	27.3	22	0.24	1.2
4307-107	AR	trans	IFM18_010	17.7	32.9	88.0	0.50	53.3	15	0.00	0.8
4307-129	Core	trans	IFM18_010	17.7	32.9	73.9	0.46	50.0	14	0.19	0.7
4307-108	AR	trans	IFM18_010	32.9	51.2	91.1	0.45	46.2	13	0.00	0.8
4307-130	Core	trans	IFM18_010	32.9	51.2	75.5	0.53	47.7	44	0.14	1.8
4307-109	AR	trans	IFM18_010	51.2	69.5	94.6	0.92	50.0	14	0.14	1.1
4307-131	Core	trans	IFM18_010	51.2	69.5	85.6	1.04	41.7	12	0.18	0.9
4307-132	Core	trans	IFM18_012	4.9	29.9	86.0	0.86	47.8	136	0.28	1.1
4307-112	AR	trans	IFM18_012	10.4	29.9	93.0	1.15	38.9	190	0.29	1.2
4307-114	AR	trans	IFM18_025	10.1	26.8	89.4	0.66	33.3	9	0.15	2.5
4307-133	Core	trans	IFM18_025	10.1	39.0	80.9	0.47	37.5	16	0.11	0.8
4307-115	AR	trans	IFM18_025	26.8	39.0	86.9	0.61	54.3	116	0.67	1.2
4307-116	AR	trans	IFM18_026A	14.5	29.4	87.2	1.95	57.3	124	0.29	1.3
4307-134	Core	trans	IFM18_026A	14.5	29.4	85.3	2.31	62.0	79	0.10	1.3
4307-099	AR	unox	IFM18_001A	292.0	313.3	72.1	1.54	25.7	249	0.15	1.0
4307-125	Core	unox	IFM18_001A	292.0	313.3	84.3	0.70	19.0	274	0.28	0.4
4307-102	AR	unox	IFM18_003	109.1	119.5	19.0	0.42	25.0	4	0.07	2.1
4307-103	AR	unox	IFM18_003	122.8	147.4	39.1	0.46	20.0	5	0.22	1.6
4307-128	Core	unox	IFM18_003	122.8	132.0	13.9	0.36	25.0	4	0.31	1.0
4307-135	Core	unox	IFM18_003	122.8	161.8	35.9	0.64	20.0	10	0.15	2.0
4307-104	AR	unox	IFM18_003	147.4	161.8	39.4	0.66	25.0	8	0.45	1.7
4307-105	AR	unox	IFM18_004	122.5	131.7	52.7	1.12	36.4	11	0.07	2.5
4307-106	AR	unox	IFM18_004	182.3	191.4	34.4	0.32	42.9	7	0.15	2.5
4307-110	AR	unox	IFM18_010	100.0	113.7	46.2	0.39	50.0	2	0.37	1.3
4307-111	AR	unox	IFM18_010	131.1	149.4	60.5	0.43	66.7	3	0.22	1.0
4307-113	AR	unox	IFM18_012	110.6	119.8	74.5	0.47	33.3	12	0.74	2.2
4307-117	AR	unox	IFM18_026A	89.0	95.6	50.0	0.46	50.0	4	0.29	1.6
4307-118	AR	unox	IFM18_026A	106.1	111.9	54.0	0.50	41.2	51	0.59	3.9

Bottle-roll test results indicated good potential for heap leaching of the Florida Mountain transitional type material. Gold recoveries generally ranged from 60% to 90%, and averaged 82%, in 96 hours of leaching. Corresponding silver recoveries averaged 46%. Gold and silver recovery rates were moderate, and it was generally the case that gold and silver extraction were progressing at a slow rate when leaching was ended. These results indicate that recoveries would improve with longer leaching

cycles. For the transitional material, cyanide consumption of 0.19 kg NaCN/t on average, and lime requirement of 1.2 kg/t on average, were low.

The unoxidized composites were not nearly as amenable to agitated cyanidation treatment, at the -1.7mm feed size. With the exception of the unoxidized material from drill hole IFM18_001A, gold recoveries obtained in 96 hours of leaching generally were less than 55%. Average gold and silver recoveries from the unoxidized composites averaged 48.3% and 34.3%, respectively. These results indicate poor potential for heap leaching of the Florida Mountain unoxidized materials.

Only two oxide composites were tested. Both showed high gold recovery by agitated cyanidation at the -1.7mm feed size, indicating good potential for heap leaching.

Seven select core composites were used for column-leach tests, at an 80% -13mm (0.5 inch) feed size to evaluate heap leach amenability. One composite was prepared from a mix of oxide and transitional material, four were prepared from transitional material only, one was prepared from a mix of transitional and unoxidized material, and one composite consisted of unoxidized material. Unoxidized material generally was not subjected to column testing because of the poor bottle-roll test recoveries from this material type. The column-test charges were leached without agglomeration, and a small quantity of lime (0.7 - 1.8 kg/t) was added before leaching, for pH control. Leaching conditions included solution application at a rate of 9.8 Lph/m^2 at a cyanide concentration of 1.0 g NaCN/L for leach cycles ranging from 63 to 97 days. Summary results from the column-leach tests are shown in Table 13.17. Gold leach rate data for the oxide and transitional material type samples are shown in Figure 13.7.

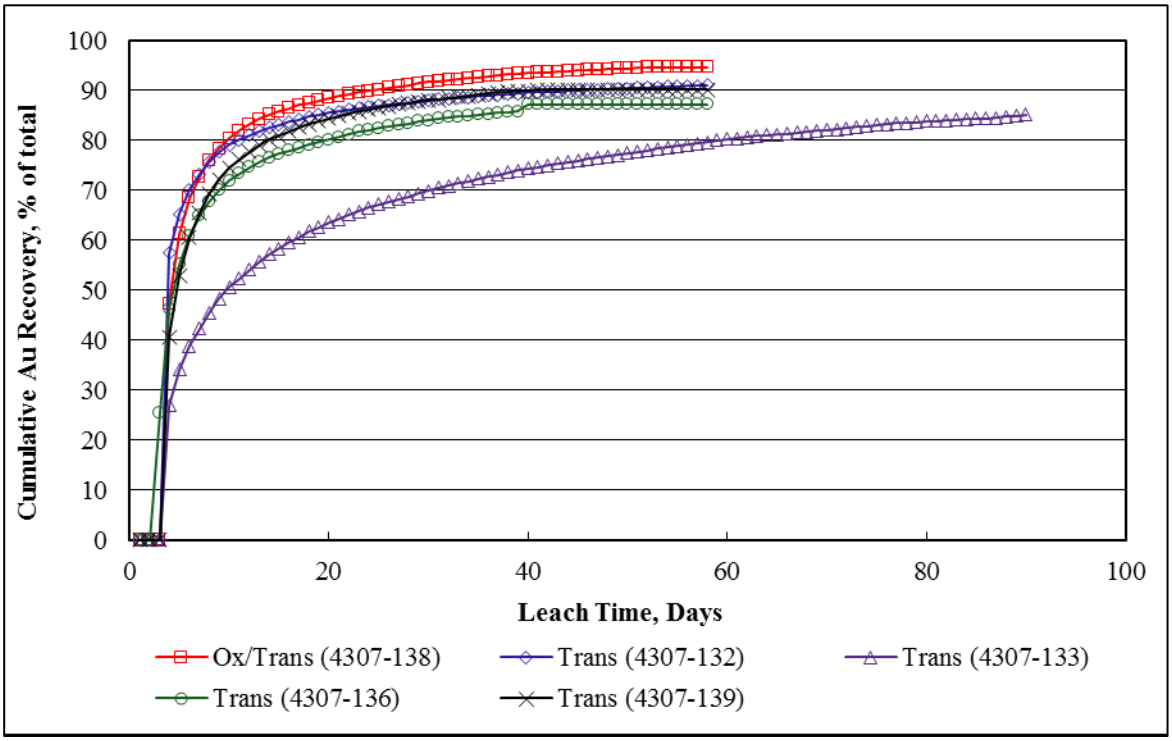
Table 13.17 Florida Mountain 2018-2019 Column-Leach Results

(Drill Core Composites, 80%-12.5mm Feed Size)

Composite	Drill Hole	Depth, feet	Oxidation Class	Leach Time, Days	Au Recovery, %	Head Grade, g Au/t	Ag Recovery, %	Head Grade, g Ag/t	Reagent Requirements, kg/t ore	
									NaCN Cons.	Lime Added
4307-138	IFM18_003	0-108', 143-233.5'	mixed (ox/trans)	65	94.7	0.75	37.5	16	1.16	1.0
4307-132	IFM18_012	16-98'	trans	63	91.3	0.92	43.3	67	1.29	1.0
4307-133	IFM18_025	33-128'	trans	97	85.5	0.69	39.0	59	3.08	0.7
4307-136	IFM18_001A	38.5-66.5', 128-185'	trans	63	87.2	0.39	41.3	75	1.17	1.1
4307-139	IFM18_010	58-228'	trans	65	90.2	0.61	26.3	19	1.18	1.0
4307-137	IFM18_001A	218-268', 958-1028'	mixed (trans/unox)	97	65.7	1.02	30.0	170	2.03	0.5
4307-135	IFM18_003	403-531'	unox	64	30.0	0.60	10.0	10	1.22	1.8

Note: Ox denotes oxide, trans denotes transitional and unox denotes unoxidized.

Figure 13.7 Florida Mountain 2018-2019 Gold Leach Rate Profiles for Column-Leach Tests
(Ox and Trans Drill Core Composites; 80% -12.5mm Feed Size)



The Florida Mountain column results showed a very high gold recovery of 94.7% for the mixed oxidized-transitional composite. Gold recoveries from the transitional composites ranged from 85.5% to 91.3% and averaged 88.5%. The mixed unoxidized-transitional composite gave a significantly lower gold recovery of 65.7%, and the unoxidized material composite gave a poor gold recovery of 30.0%. Silver recoveries for the oxide and transitional composites ranged from 26.3% to 43.3% and averaged 37.5%. The unoxidized composite gave a lower silver recovery (10.0%). Column-test cyanide consumptions varied with leach cycle duration, as is commonly observed with laboratory column tests. Cyanide consumptions for the composites leached for 63 to 65 days ranged from 1.16 to 1.29 kg NaCN/t. Cyanide consumptions for the two composites leached for 97 days ranged from 2.03 to 3.08 kg NaCN/t. The 0.5 to 1.8 kg/t lime, added before leaching, was sufficient for maintaining leaching pH.

A fixed-wall hydraulic conductivity (load/permeability) test was conducted on a composite from five of the seven leached column residues. The two longer-term leach tests were not included in the composite. Results showed very high hydraulic conductivity (2.0×10^{-1} cm/s) at the 100-meter simulated heap stack height. These results indicate the Florida Mountain transitional material will remain adequately permeable at commercial heap stack heights of up to 100 meters.

13.4.2.3 Florida Mountain Agitated Cyanide Leach Testing

Ten of the 2018-2019 Florida Mountain unoxidized composites, along with one of the transitional composites were used for bottle-roll leach tests at an 80% -75µm (200 mesh) feed size to evaluate amenability to grind-leach (cyanidation) treatment. Summary leaching conditions and results from those tests are shown in Table 13.18.

Table 13.18 Florida Mountain 2018-2019 Whole Ore" Milling & Cyanidation Tests
(80%-75µm Feed Size, 72 Hour Leach (with interim sampling), 40% Solids, 1.0 g NaCN/L)

Sample Description						Au Recovery, %	Head Grade, g Au/t	Ag Recovery, %	Head Grade, g Ag/t	Reagent Requirements, kg/t ore	
Drill Composite	Type	Oxidation	Hole	Interval, meters from	Interval, meters to					NaCN Cons.	Lime Added
4307-155	AR	trans	IFM18_001A	39.0	87.5*	94.4	0.54	92.5	29	0.17	1.7
4307-099	AR	unox	IFM18_001A	292.0	313.3	96.1	1.54	32.7	275	0.20	1.3
4307-135	Core	unox	IFM18_003	122.8	161.8	81.0	0.58	52.8	11	0.16	1.9
4307-156	AR	unox	IFM18_003	109.1	161.8*	76.8	0.56	53.6	6	0.43	2.0
4307-140	Core	unox	IFM18_004	122.5	131.7	87.8	0.98	48.3	6	0.18	3.1
4307-141	Core	unox	IFM18_004	147.2	188.4	81.6	0.49	61.5	8	0.08	2.7
4307-157	AR	unox	IFM18_004	122.5	191.4*	79.7	0.59	66.3	10	0.48	2.5
4307-142	Core	unox	IFM18_010	131.2	169.5	89.8	0.59	63.0	5	0.22	2.2
4307-158	AR	unox	IFM18_010	100.0	149.4*	89.5	0.38	>66.7	<3	<0.07	1.8
4307-159	AR	unox	IFM18_026A	89.0	111.9*	81.8	0.33	79.2	24	0.38	3.5
4307-143	Core	unox	IFM18_026A	89.0	108.8	93.0	0.57	90.8	51	0.31	2.6

* *Non-continuous intervals.*
Note: AR denotes assay reject composites. Core denotes split drill core composites.

All of the unoxidized Florida Mountain composites tested were amenable to grind-leach treatment, at an 80% -75µm feed size. Gold recoveries ranged from 76.8% to 96.1%, and averaged 85.7%, in 72 hours of leaching. Corresponding silver recoveries ranged from 32.7% to 90.8% and averaged 61.5%. Reagent consumptions were fairly low.

The single Florida Mountain transitional composite tested gave very high recoveries of 94.4% gold and 92.5% silver. Further testing of Florida Mountain transitional material for grind-leach and gravity concentration/tailings cyanidation processing will be required for a trade-off study against heap leaching.

13.4.2.4 Florida Mountain Gravity Concentration and Treatment of Gravity Tailings

Considering the consistent behavior of the Florida Mountain unoxidized composites, a single master composite was prepared for comparative gravity concentration with evaluation of gravity-tailings cyanidation and gravity-tailings flotation. The resulting flotation concentrate was also subjected to regrinding followed by intensive cyanidation testing. The composite (designated 4307-160) was a "master composite" prepared from unoxidized composites 4307-135, 140, 141 and 142, which were comprised of drill core from holes IMF18_003, 004 and 010.

A single gravity-concentration test was conducted on the Florida Mountain unoxidized master composite (4307-160) to evaluate response to gravity concentration and to generate gravity tailings for cyanidation and flotation testing. The resulting cleaner concentrate was assayed. The cleaner tails and rougher tails

were recombined and then split to obtain feeds for cyanidation and flotation testing. The gravity cleaner was 0.04% of the feed weight, assayed 148 g Au/t and 316 g Ag/t, and was not included in the agitated cyanidation or flotation test feeds.

Agitated cyanidation tests were conducted on gravity tailings generated as described in the preceding paragraph, at five tailings regrind sizes ranging from 80% -150µm to 80% -45µm. Bulk sulfide flotation tests were also conducted on separate splits of the same gravity tailings, at regrind sizes ranging from 80% -212µm (no regrind) to 80% -75µm, to evaluate the potential for upgrading the gravity tailings by flotation. Summary results from the gravity/cyanidation tests are shown in Table 13.19. Summary results from the gravity/flotation tests are shown in Table 13.20.

Table 13.19 Florida Mountain 2018-2019 Gravity Concentration - Agitated Cyanidation of Gravity Tailings

(Comp. 4307-160, 80%-212µm Primary Grind Size, Reground Gravity Tailings, 72 Hour Leach, 40% Solids, 1.0 g NaCN/L)

Cyanide Leach Feed Regrind Size	Au Recovery, % of total			Head Grade, g Au/t	Ag Recovery, % of total			Head Grade, g Ag/t	Reagent Requirements, kg/t ore	
	Gravity[1)]	CN[2)]	Combined[3)]		Gravity[1)]	CN[2)]	Combined[3)]		NaCN Cons.	Lime Added
80%-150µm	7.5	74.3	81.8	0.79	1.8	55.2	57.0	6.9	0.04	1.6
80%-106µm	7.5	74.0	81.5	0.79	1.7	56.2	57.9	7.4	0.10	1.6
80%-75µm	6.8	75.4	82.2	0.87	1.7	58.4	60.1	7.5	0.11	1.6
80%-53µm	7.9	74.2	82.1	0.75	1.6	59.8	61.4	7.7	0.12	1.8
80%-45µm	7.8	73.1	80.9	0.76	1.7	69.6	71.3	7.2	0.02	1.9

1) Recovery to a gravity concentrate produced with a 0.04% mass pull, with a grade of 148 gAu/mt and 316 gAg/mt.

2) Recovery by agitated cyanidation of the gravity tailings (99.96% mass fraction).

3) Combined recovery by gravity concentrate and extraction during agitated cyanidation of gravity tailings.

Table 13.20 Florida Mountain 2018-2019 Gravity Concentration with Flotation of Gravity Tailings

(Comp. 4307-160, 80%-212µm Gravity Concentration Feed)

Regrind Size	Cleaner Concentrate (Gravity + Flotation Cleaner)					Rougher Concentrate (Grav. Cleaner + Flotation Rougher)				
				Recovery					Recovery	
	Mass, %	g Au/t	g Ag/t	% Au	% Ag	Mass, %	g Au/t	g Ag/t	% Au	% Ag
80%-212µm[1)]	4.3	17.8	146	96.0	83.8	6.6	11.80	99	97.6	87.5
80%-180µm	3.6	18.1	185	91.2	82.5	8.1	8.31	88	94.9	88.7
80%-150µm	3.2	25.0	180	86.7	75.8	8.2	11.01	83	96.1	88.0
80%-106µm	4.4	17.3	167	92.7	84.3	11.2	6.95	69	95.7	89.8
80%-75µm	2.7	27.7	204	82.4	76.2	10.8	7.51	58	90.1	87.5

1) Gravity concentration feed size was 80%-212µm. No regrind before flotation.

2) Includes gravity cleaner concentrate (0.04% mass pull, 148 gAu/mt and 316 gAg/mt), and flotation cleaner concentrate.

The gravity tailings were amenable to agitated cyanidation treatment at the regrind sizes tested. Combined gold recovery (gravity concentration + tailings cyanidation) ranged from 80.9% and 82.2% and was not sensitive to regrind size. Combined silver recoveries increased with decreasing regrind size, from 57.0% to 71.3%. Reagent consumptions were low.

The gravity tailings also responded very well to bulk sulfide flotation. The combined concentrates (gravity cleaner concentrate + flotation rougher concentrate), produced at regrind sizes of as fine as 106µm, were equivalent to between 6.6% and 11.2% of the "whole-ore" mass, and contained 94.9% to 97.6% of the gold and between 87.5% and 89.8% of the silver contained in the "whole-ore". Although mass pull to the flotation concentrate tended to increase with decreasing regrind size, recoveries did not increase. Sulfide sulfur recovery by flotation ranged from 86.4% to 90.4%. Gold recovery (90.1%) and sulfide sulfur recovery (76.9%) were somewhat lower at the 75µm regrind size.

13.4.2.5 Florida Mountain Flotation Concentrate Regrind/Agitated Leach

Based on the positive results obtained from the flotation testing conducted on the Florida Mountain unoxidized material, a larger, 8-kilogram flotation test was conducted on the same gravity tailings (unoxidized master composite 4307-160), at an 80% -212µm feed size (no regrind) to generate rougher concentrate for cyanidation testing. The purpose for this test was to determine if, by fine regrinding of the flotation concentrate, cyanidation recoveries could be improved beyond those observed during agitated cyanidation testing on the same gravity tailings. The flotation rougher concentrate produced was used as feed for an intensive cyanidation test. Summary test conditions (flotation concentrate leach) and summary gravity/flotation/concentrate cyanidation test results are summarized in Table 13.21. Concentrate gold and silver leach rate data are shown in Figure 13.8.

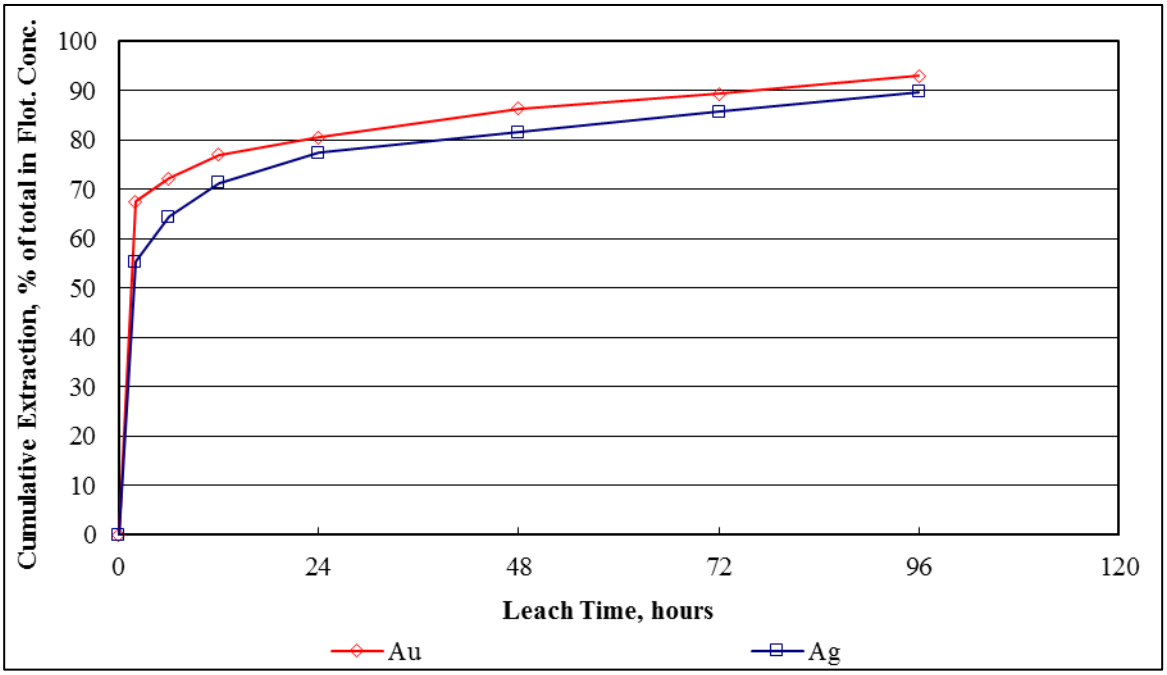
Table 13.21 Florida Mountain 2018-2019 Gravity Concentration, Flotation of Gravity Tailings, Regrind Leach of Flotation Concentrate

(Unoxidized Master Composite 4307-160, 80% -212µm Feed Size; 95%-37µm Flotation Concentrate Regrind, 96 Hour Leach, 25% Solids, 5.0 g NaCN/L)

								Reagent Requirements, kg/t ore	
	Weight,	Assay or Calculated Grade			Distribution			NaCN	Lime
Product	%	g Au/t	g Ag/t	% S=	% Au	% Ag	% $S^{=}$	Consumed	Added
Gravity Cl. Conc.	0.04	148	316	N/A	8.8	1.5			
Flotation Ro. Conc.	4.58			15.6			84.3		
Recovered, CN		11.81	150		80.9	78.7		0.14	0.2
Leach Tail, CN		0.88	17		6.0	8.9			
Combined Recovery[1)]					**89.7**	**80.2**			
Flot. Ro. Tail	95.38	0.03	1	0.14	4.3	10.9	15.7		
Combined Tail	99.96	0.07	2	0.85	10.3	19.8			
Composite	**100.00**	**0.67**	**9**	**0.85**	**100.0**	**100.0**	**100.0**	**0.14**	**0.2**

1) Includes gold and silver reporting to the gravity cleaner concentrate and extracted by cyanidation of the reground flotation concentrate.

Figure 13.8 Florida Mountain 2018-2019 Gold and Silver Leach Rates, Agitated Cyanidation of Flotation Rougher Concentrate
(Comp. 4307-160, 95% -37µm Regrind Size)



The flotation rougher concentrate produced from the Florida Mountain unoxidized master composite 4307-160 was readily amenable to agitated cyanidation treatment, at a 95% -37µm regrind size. Gold and silver recoveries were 93.1% and 89.8%, respectively, from the flotation concentrate. The combined recoveries by gravity concentration (cleaner concentrate) and cyanidation of the flotation concentrate were equivalent to 89.7% of the gold and 80.2% of the silver contained in the "whole-ore" feed. Cyanidation recovery rates were moderate, and extraction of gold and silver was progressing at a slow, but significant rate when leaching was terminated after 96 hours. Optimization of the leaching cycle and leaching conditions will likely lead to incrementally higher recoveries. Cyanidation reagent consumptions were very low and equivalent to only 0.14 kg NaCN/t and 0.2 kg lime/t on a "whole-ore" mass basis.

Recoveries obtained by regrind and cyanidation of the flotation concentrate compare favorably to those obtained by gravity concentration with cyanidation of the gravity tailings. Gold and silver recoveries obtained from the master composite by gravity concentration with cyanidation of the gravity tailings did not exceed 82.2% and 71.3%, respectively, at gravity tailings regrind sizes of as fine as 80%-45µm. These results indicate apparent increases in overall gold and silver recoveries of approximately 7% and 9%, respectively, were obtained by very fine regrinding of the flotation concentrate. The test results demonstrate that a relatively coarse primary grind size, with correspondingly lower grinding costs, will be possible for the Florida Mountain unoxidized material.

13.5 Summary Statement

Available test data indicates that the oxide and transitional materials from both the DeLamar and Florida Mountain deposits behave reasonably similarly and can be processed by heap-leach cyanidation. Because higher clay content may be encountered for some zones in the DeLamar deposit, agglomeration pre-treatment may be required for those materials, and is included in the PEA for heap leaching of the DeLamar oxide and transitional mineralization. As shown by current testing, other DeLamar oxide and transitional materials will likely not require agglomeration pretreatment. Further studies are planned to characterize the DeLamar heap-leach feed for clay content and the need for agglomeration. Low to moderate cyanide consumptions are indicated for heap leaching of the oxide and transitional material types. Lime or cement demand is expected to be variable.

Improvements in silver recoveries and, to a lesser degree, in gold recoveries can likely be achieved by grinding and agitated leaching of the DeLamar and Florida Mountain oxide and transitional material types. Once sufficient metallurgical test data are available, trade-off studies will be required to evaluate agitated leaching versus heap leaching for these materials.

Unoxidized materials from the Florida Mountain deposit was not amenable to heap leaching. This material type responded well to upgrading by gravity concentration, and by flotation, as well as to grind-leach. Testing has shown that the highest gold and silver recoveries were obtained from the Florida Mountain unoxidized materials by gravity concentration, followed by flotation of the gravity tails, with regrinding and agitated cyanide leaching of the flotation concentrate. This process envisions no further processing of the flotation tailings for metal recovery.

Unoxidized materials from the DeLamar area deposits appear not to be amenable to heap leaching. Unoxidized material from the DeLamar deposit generally responds well to upgrading by gravity concentration and flotation. The most likely processing option for this material will include milling followed by flotation (possibly also with gravity concentration) to produce a concentrate. Possible processing options for gold and silver recovery from the concentrate include regrind followed by agitated cyanidation, shipment off site for toll processing; or on-site oxidative treatment (such as pressure oxidation, roasting, or bio-oxidation), followed by agitated cyanidation of the oxidized concentrate. Further testing is in progress for the evaluation of these processing options. It was also noted that a small but significant portion of the unoxidized (particularly Sullivan Gulch) DeLamar samples tested exhibited moderate amenability to grind-leach processing. Further testing, mineralogy and geo-metallurgical modelling will be required to better understand the factors controlling the observed variability in response of the DeLamar unoxidized materials to the various processing methods considered.

Selected processing methods, along with the corresponding estimate recoveries and reagent consumptions for the DeLamar and Florida Mountain oxide and transitional material types, along with the Florida Mountain unoxidized material type are shown in Table 13.22.

The DeLamar unoxidized material was not included in the current PEA. Further mineralogical and metallurgical testing, as well as geo-metallurgical modelling is required and planned to characterize that material to the degree required for a PEA. Evaluation of processing options for this material type is ongoing, with the most likely process alternatives as described above.

Table 13.22 DeLamar Project 2019 PEA Recovery and Reagent Estimates

Parameter	DeLamar	Florida Mountain
Heap Leach - Oxide	Crush/Agglomerate/Heap Leach	Crush/Heap Leach
Crush Size	80%-13mm	80%-38mm
Au Recovery	80%	90%
Ag Recovery	30%	40%
NaCN, kg/t	0.3	0.4
Lime, kg/t	---	1.0
Cement, kg/t	3.0	---
Heap Leach - Transitional	Crush/Agglomerate/Heap Leach	Crush/Heap Leach
Crush Size	80%-13mm	80%-38mm
Au Recovery	75%	85%
Ag Recovery	30%	40%
NaCN, kg/t	0.4	0.4
Lime, kg/t	---	1.0
Cement, kg/t	3.0	---
Milling - Unoxidized Material Type	Not included in PEA	Grind/Gravity/Flotation/Conc. Leach
Primary Grind Size		80%-212µm
Au Recovery		90%
Ag Recovery		80%
Concentrate Mass (for regrind)		5%
Regrind Size		95%-37µm
Potassium Amyl Xanthate (PAX), kg/t		0.025
AERO 208 (Dithiophosphate), kg/t		0.050
AEROFROTH 65 (Polyglycol Frother), kg/t		0.1
NaCN Consumption, kg/t		0.2
Lime Consumption, kg/t		0.2

Mr. McPartland has reviewed the historical metallurgical studies and concludes the information provides a useful context from which to develop additional metallurgical programs. While the available historical information is not sufficient to allow for an assessment of the representativity of the samples tested, samples used in the 2018-2019 metallurgical program are reasonably representative considering both the stage of the project development and the magnitude of the testing completed as of the effective date of this report. However, further testwork of samples collected from additional portions of the project will be needed as the project advances. Other than as discussed herein, the author is not aware of any processing factors or deleterious elements that could have a significant effect on potential economic extraction.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The mineral resource estimations for the DeLamar project, which includes the DeLamar and Florida Mountain deposits, were completed for public disclosure in accordance to the guidelines of NI 43-101. The modeling and estimation of the mineral resources were completed under the supervision of Michael M. Gustin, a qualified person with respect to mineral resource estimations under NI 43-101. Mr. Gustin is independent of Integra by the definitions and criteria set forth in NI 43-101; there is no affiliation between Mr. Gustin and Integra except that of independent consultant/client relationships.

This report presents updated gold and silver resources for the DeLamar and Florida Mountain deposits that have an effective date of May 1, 2019. No mineral reserves have been estimated for the DeLamar project.

The DeLamar project resources are classified in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories in accordance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014) and therefore NI 43-101. CIM mineral resource definitions are given below, with CIM's explanatory text shown in italics:

> **Mineral Resource**
>
> *Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.*
>
> A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
>
> *Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.*
>
> *The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic*

extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and

testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

14.2 DeLamar Project Data

The DeLamar project gold and silver resources were estimated using drill data generated by Integra through April 2019, as well as the data derived from the exploration programs of the various historical operators discussed in Section 10.0. This information, including the data derived from RC, conventional rotary, and diamond-core drill holes, current topography, historical documentation of the as-mined open-pit topographies, cross-sectional lithological and structural interpretations, and documentation of historical underground workings, were provided to MDA by Integra.

14.2.1 Drill-Hole Data

The historical project data used mine-grid coordinates, a local grid system in Imperial units developed in the early 1970s and used throughout the life of the DeLamar open-pit mining operations. The original down-hole drill intervals were in feet, and the gold and silver analyses were primarily reported in ounces per ton. In 2018, Integra completed a LiDAR aerial survey of the entire DeLamar project area, obtained historical survey data in both mine-grid and real-world coordinates, and transformed the drill-hole locations into UTM Zone 11 NAD 83 coordinates with the assistance of MDA. All project down-hole drill depths, assays, and geologic logging intervals were then converted into meters and grams-per-tonne.

As discussed in Section 11.0, the historical exploration and development drill-hole samples were variably analyzed for gold and silver by fire assay and AA methods, and for a period of time the mine-lab silver AA values were factored to account for incomplete sample digestions. The historical DeLamar and Florida Mountain databases that supported the open-pit mining operations included these various types of analyses, with multiple analytical types commonly completed on a single sample interval. The databases also included "FFAU" and "FFAG" fields that were comprised of the gold and silver values, respectively, used in all mine-site purposes, including historical resource and reserve estimations. The FFAU and FFAG values prioritized fire assays, completed by the mine site or outside laboratories, over AA analyses. The factored AA silver values were in the FFAG field, while the original, unfactored AA silver analyses were also retained in the mine-site databases.

After auditing the historical data, MDA constructed independent resource databases for the DeLamar and Florida Mountain areas. Gold and silver values used in the resource estimations discussed herein were prioritized as follows: fire assays by outside labs were given top priority, followed by fire assays by the on-site mine lab, with AA analyses used only where no other data were available. No factored AA silver values reside in MDA's resource databases. However, the unfactored historical AA silver analyses were not used in the resource estimations, as these analyses demonstrably understate silver grades (Section 12.2.1).

As discussed in Section 10.7, drill intervals identified as having significant sample quality issues, including down-hole contamination, were excluded from use in the resource estimation. In addition, colluvial deposits were either explicitly excluded from the gold-and silver-domain modeling described below, as was commonly the case for the DeLamar deposit, or tagged for exclusion directly in the project databases, as was the case for the Florida Mountain database, where significantly mineralized colluvium was frequently intersected in the top few meters of drill holes.

14.2.2 Topography

Integra provided MDA with project-wide elevation data from their LiDAR survey, which was used to create digital topographic surfaces for both the DeLamar and Florida Mountain deposit areas. This current surface reflects post-mining reclamation, including re-contouring of waste dumps and the partial backfilling of many of the open pits.

Integra also provided MDA with original historical paper plots of final post-mining topographies of the open pits at both the Florida Mountain and DeLamar areas. MDA used these paper plan maps to create digital 'as-mined' topographic surfaces that encompasses the areas of historical open-pit mining. Based on other historical data, including blast-hole information, as well as the current topography derived from the LiDAR survey, Mr. Gustin believes the modeled as-mined surfaces reasonably locate the volumes mined during the historical open-pit operations.

14.2.3 Modeling of Historical Underground Workings

Integra provided MDA with three-dimensional digital linework created by Kinross that represents historical drifts, crosscuts, and developmental workings in the DeLamar area. This modeling by Kinross, which was based on historical records reviewed by the authors, indicates that the historical underground workings in the DeLamar area lie almost entirely inside of the historical North DeLamar and Sommercamp open pits. However, the drifts along the mined vein structures and related developmental winzes were useful in the modeling of the unmined gold and silver resources lying below and adjacent to the pits, as they provided evidence of the strikes and dips of the mined mineralized structures.

Underground workings at Florida Mountain, including drifts, cross cuts, winzes, shafts, and stopes, are documented by a series of original hand-drafted level plans, long sections, and cross sections in the possession of Integra that date from the late 1800s to the early 1900s. MDA used these drawings to create three-dimensional digital models of the underground workings and stopes, although there is little information as to the widths of the stopes. While these drawings are unlikely to include all historical underground mining that took place at Florida Mountain, there is good evidence that a high percentage of the stopes from the Black Jack – Trade Dollar workings are represented.

14.3 Geological Modeling

Integra completed hand-drawn lithological and structural interpretations on a set of paper cross sections that span the extents of the Florida Mountain and DeLamar resource areas, except for the Milestone area. MDA digitized these cross sections and used them as the base for modeling the gold and silver mineral domains discussed in Section 14.8.1. Lithological contacts that influenced the distributions of the gold and silver mineralization, as well as faults modeled on sections by Integra and high-angle mineralized zones modeled by MDA, were represented as three-dimensional wireframe surfaces that served as guides for the detailed modeling of the gold and silver mineralization.

14.4 Deposit Geology Pertinent to Resource Modeling

The DeLamar area mineralization is predominantly influenced by restricted high-angle zones of higher-grade mineralization and much larger bodies of shallowly dipping mineralization. The former occurs along faults identified by Integra's geologic team and presumed faults, with or without demonstrable vertical offsets, interpreted by MDA during the resource modeling. The broad low-angle mineralization is hosted in felsic volcanic units that lie above the lower basalt and below the banded rhyolite. The low-angle morphology roughly mimics the attitude of these felsic units; this is likely in part due to the permeation of fluids along volcanic lithologies, but also reflects ponding of fluids along the basal contact of the banded rhyolite, which is often characterized by a clay zone. While only minor

mineralization has been drilled within the lower basalt, low-grade mineralization does occur locally within the banded rhyolite.

At Florida Mountain, the gold and silver mineralization drilled to date also occurs primarily within felsic volcanic units that overlie basaltic flows. Much of the modeled mineralization remains open at depth. The basalt and underlying Cretaceous granodiorite host most of the high-grade veins that were the focus of the historical underground mining at Florida Mountain. At the elevations drilled to date, the Florida Mountain mineralization occurs along multiple, broad, north- to northwest-striking zones with steep dips in either direction. From west to east, these zones are centered on the historical Ontario, Tip Top, Arcuate, Alpine, Stone Cabin, and Trade Dollar-Black Jack mining and exploration areas. In detail, each of these mineralized zones are comprised of complex networks of thin, interweaving mineralization that forms what can be considered large-scale stockwork zones. Taken as a whole, these zones formed bulk-mineable bodies. There are some indications that the zones may be pinching down with depth, and perhaps will coalesce into discreet structures, although this cannot be demonstrated without further drilling.

MDA reviewed the distribution of the silver mineralization intersected in drilling carefully, especially in the silver-rich DeLamar area, in an effort to discern the presence (or absence) of potential supergene-enriched zones, which would be important to the resource modeling. Only a few limited areas were found that might be suggestive of supergene enrichment, but the evidence was not conclusive. Several historical reports state that secondary enrichment of silver probably occurred on a limited scale, although the evidence cited is restricted to the presence of cerargyrite.

14.5 Water Table

The 1974 historical feasibility study, which focused on the Sommercamp and North DeLamar areas, stated that surface oxidation generally does not extend deeper than about 55 meters from the surface, except along fault zones (Earth Resources Company, 1974). The water table was stated in the 1974 study to lie at an elevation of approximately 1,845 meters, considerably deeper than the level of oxidation. These statements presumably applied only to the two deposit areas that were the subject of the feasibility study. A later mine document reported a water table depth of 1,810 meters at the north end of the Sommercamp – Regan zone, which at the time included what was referred to as South Wahl (Pancoast, 1990). Ms. Richardson indicated to Mr. Gustin and Mr. Weiss that the water table lies near the bottoms of the North DeLamar and South Wahl pits, at elevations of about 1,820 and 1905 meters, respectively. The authors have not reviewed substantive information on groundwater in the Florida Mountain area, but it is believed to be below the depth of most of the historical drill holes.

No modeling of the water table has been undertaken at the DeLamar project.

14.6 Oxidation Modeling

Due to the importance of oxidation state on metallurgical processes, Integra undertook comprehensive logging of oxidation using historical RC / rotary chipboards present at the project site. MDA then used the Integra oxidation logs to create wireframe solids of oxidized and unoxidized zones, which were used to code the DeLamar and Florida Mountain block models. Model blocks lying between those coded as oxidized and unoxidized were coded as transitional.

Metallurgical results from testing of samples from the three oxidation zones modeled at the DeLamar project were critical in the selection of potential processing options for each of the two deposits, which in turn played a role in the determination of the resource cutoff grades applied by oxidation zone (discussed further below).

14.7 Density Modeling

A number of references to density values were reviewed in the available historical records, including some density studies with limited numbers of actual density determinations listed. These datasets are generally only partially documented, with many lacking a description of the determination methods. While the methodologies used for the density determinations are often unclear, the records indicate determinations were done by a variety of methods, including water displacement, water immersion, volume/weight, and nuclear methods.

MDA compiled the data from two of the more completely documented historical specific gravity ("SG") studies of samples from the DeLamar area. The 13 measurements yield an average SG of 2.31. Historical DeLamar and Florida Mountain resource and reserve estimations undertaken during the open-pit operations most commonly used a global tonnage factor (mineralized and unmineralized rock) of 13.5 ft^3/ton, which equates to an SG of 2.37. The historical open-pit operation used a wet density of 13.5 ft^3/ton throughout the life of the mine to determine mill-feed tonnages and waste. Based on various measurements, the mine assumed 7.5% moisture in the mined materials at DeLamar and 6% at Florida Mountain. These values equate to global (dry) SGs of 2.21 for DeLamar and 2.24 for Florida Mountain. A total of 12 historical SG determinations from Florida Mountain drill core compiled by Mr. Gustin average 2.41.

Integra routinely measured the SG of selected samples of its drill core using the water immersion method. Table 14.1 summarizes Integra's SG results for samples of core from holes drilled at the DeLamar deposit that are included in the current resource database. The results are compiled by oxidation state and whether the samples are within the gold and/or silver mineral domains that constrain the resource estimations (mineralized samples) or lie outside of the domains

Table 14.1 Integra Specific Gravity Determinations from DeLamar Deposit Drill Core

Oxidation	Au-Ag Domain	Mean	Median	StdDev	CV	Min	Max	Count
oxidized	outside	2.14	2.18	0.2	0.09	1.59	2.52	39
oxidized	inside	2.19	2.22	0.18	0.08	1.78	2.45	11
transitional	outside	2.24	2.25	0.25	0.11	1.73	2.63	25
transitional	inside	2.37	2.42	0.5	0.21	1.74	3.18	6
unoxidized	outside	2.34	2.32	0.26	0.11	1.69	3.18	266
unoxidized	inside	2.43	2.46	0.27	0.11	1.63	3.23	73

The data show a trend of increasing SG values from oxidized to transitional to unoxidized zones, which was expected. An examination of the raw data also indicates a tendency for SG to increase as metal grades increase, which may be due to lowering clay contents and increasing quartz as grades increase.

A summary of Integra's Florida Mountain SG data is shown in Table 14.2.

Table 14.2 Integra Specific Gravity Determinations from Florida Mountain Deposit Drill Core

Oxidation	Au-Ag Domain	Mean	Median	StdDev	CV	Min	Max	Count
oxidized	outside	-	-	-	-	-	-	-
oxidized	inside	-	-	-	-	-	-	-
transitional	outside	2.44	2.46	0.16	0.07	2.10	2.66	10
transitional	inside	2.46	2.46	0.1	0.04	2.25	2.7	20
unoxidized	outside	2.54	2.56	0.13	0.05	1.95	2.88	57
unoxidized	inside	2.48	2.44	0.15	0.06	2.31	2.78	11

In consideration of the entirety of this information, SGs of 2.20, 2.30, and 2.45 were assigned to all DeLamar mineralized and unmineralized materials within the oxidized, transitional, and unoxidized zones, respectively. Values of 2.25, 2.35, and 2.50 were assigned to Florida Mountain. These values are based primarily on the Integra SG determinations and the long-term usage of the mining operations, which predominantly mined oxidized and, to a lesser extent, transitional materials.

Specific gravity values of 1.72 and 1.76 were assigned to backfill and dump materials at the DeLamar and Florida Mountain deposits, respectively.

14.8 DeLamar Area Gold and Silver Modeling

14.8.1 Mineral Domains

A mineral domain encompasses a volume of rock that ideally is characterized by a single, natural grade population of a metal that occurs within a specific geologic environment. In order to define the mineral domains at DeLamar, the natural gold and silver populations were first identified on population-distribution graphs that plot the gold-grade and silver-grade distributions of all project drill-hole assays. This analysis led to the identification of low-, medium-, and high-grade populations for both gold and silver. Ideally, each of these populations can then be correlated with specific geologic characteristics that are captured in the project database, which can be used in conjunction with the grade populations to interpret the bounds of each of the gold and silver mineral domains. The approximate grade ranges of the low-grade (domain 100), medium-grade (domain 200), and higher-grade (domain 300) domains are listed in Table 14.3.

Table 14.3 Approximate Grade Ranges of DeLamar Area Gold and Silver Domains

Domain	g Au/t	g Ag/t
100	~0.15 to ~1	~5 to ~30
200	~1 to ~6	~30 to ~200
300	> ~6	> ~200

The DeLamar gold and silver mineral domains were modeled by interpreting silver, followed by gold, polygons on a set of vertical, 30-meter spaced, northwest-looking (Az. 320°) cross sections that span the presently drilled extents of the deposit. The mineral domains were interpreted using the gold and silver drill-hole assay data, associated drill-hole lithologic codes, documented descriptions of the mineralization, the historical underground workings, and Integra's geological cross sections.

The mineral-domain modeling for the current resource estimation was aided to a significant extent by the lithologic, structural, and mineralization cross sections completed by Integra. The Integra cross sections, coupled with the closely spaced drilling throughout the resource area, guided the MDA modeling of the gold and silver mineral domains and provided a much higher level of confidence than could be obtained in the prior resource estimation.

The low-grade gold and silver domains generally encompass relatively extensive and flat-lying to gently southwest-dipping bodies lying within the various felsic volcanic units that lie between the banded rhyolite, with its basal clay zone, and the lower basalt. The morphology of this low-grade mineralization suggests that the dominant movement of mineralizing fluids was strongly influenced by the orientations of the host volcanic units.

Relatively restricted, steeply dipping zones of mineralization in the mid-grade and higher-grade domains (domains 200 and 300, respectively) occur within the low-grade mineralization. Zones of mid-grade frequently branch out from these high-angle occurrences into the low-grade mineralization, and some of these occurrences include significant volumes of shallowly dipping mid-grade mineralization.

In areas where all volcanic units from the lower basalt to the flow-banded rhyolite are preserved, the high-angle mineralization is often truncated at the basal contact of the flow-banded rhyolite. The domain 200 and 300 mineralization then extends laterally along or close to this contract, appearing to have ponded below it. The portions of the DeLamar deposit in which the high-grade domain occurs, both in the high-angle zones and, most importantly, the lower-angle zones below the flow-banded rhyolite, were preferentially mined during the historical open-pit operations. Therefore, few of such shallow occurrences of the low-angle high-grade zones remain. The most important example of significant mining in areas where the contact zone had been eroded is at the Sommercamp pit, where all but a few erosional remnants of the contact mineralization were present prior to mining. In this case, the high grades and frequency of the high-angle mineralization were sufficient to warrant its extraction.

The lower contact of the banded rhyolite, as well as the faults that are evidenced by its displacements, were modeled by Integra. This contact and the faults were used extensively in the mineral-domain modeling. Steeply dipping high-grade zones not associated with faults recognized by Integra were typically modeled by MDA as having steep southwesterly dips, which is consistent with historical underground stopes in the Sommercamp and North DeLamar areas.

The main DeLamar area mineralization, which includes the entire area of historical mining, extends continuously over a northwest strike extent of about three kilometers, a maximum northeast-southwest width of 1.2 kilometers, and an elevation range of 570 meters. The Milestone portion of the DeLamar mineralization, which lies about three-quarters of a kilometer northwest of the northwesternmost extents of the main DeLamar area, adds an additional 640 meters of strike to the resource modeling.

Cross-sections showing examples of the gold and silver mineral domains for the Sullivan Gulch, Sommercamp – North DeLamar – Regan, and Glen Silver areas of the resources are shown in Figure 14.1 through Figure 14.6.

Figure 14.1 Cross Section 1230 NW Showing Sullivan Gulch Gold Domains

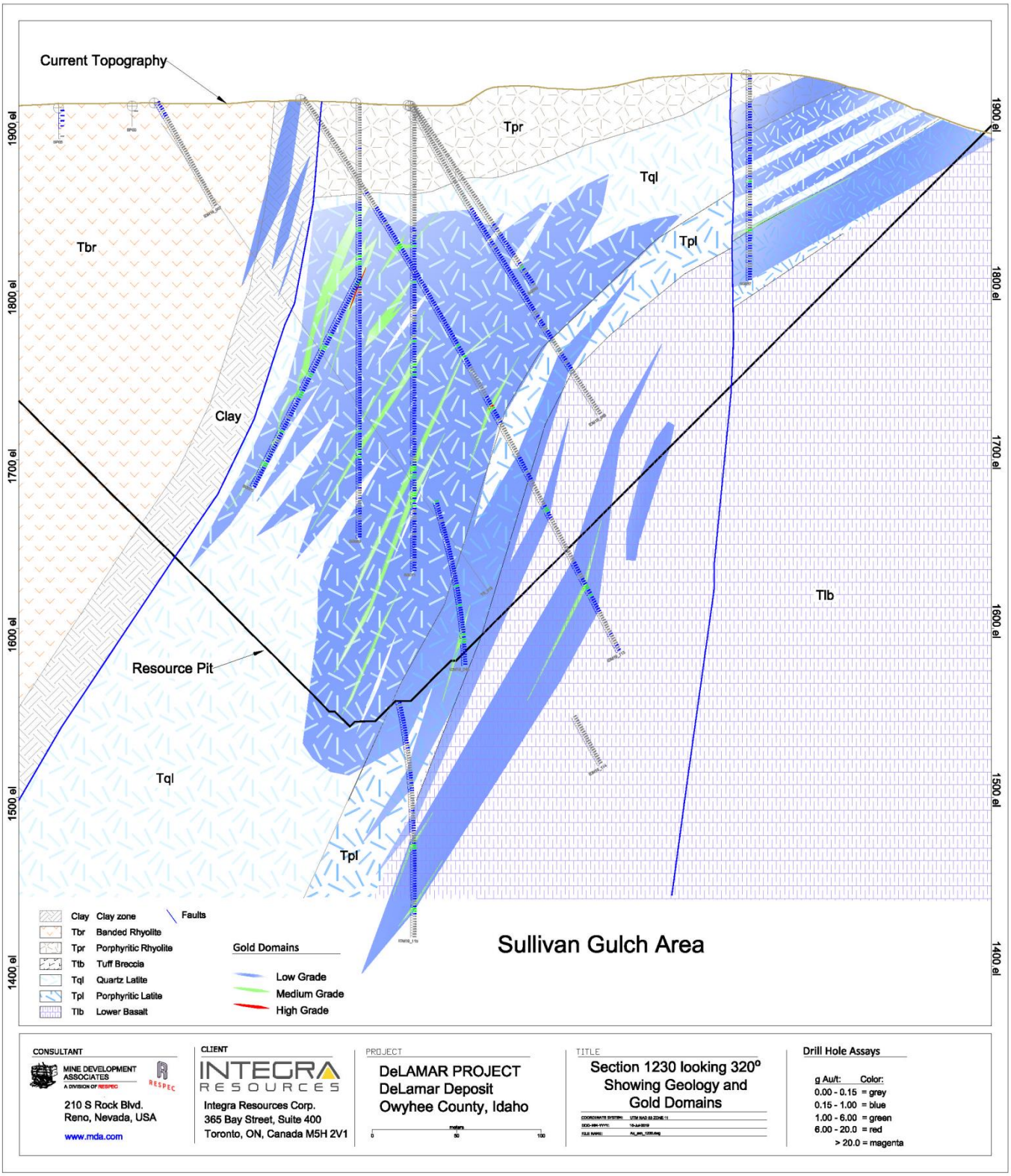


Figure 14.2 Cross Section 1230 NW Showing Sullivan Gulch Silver Domains

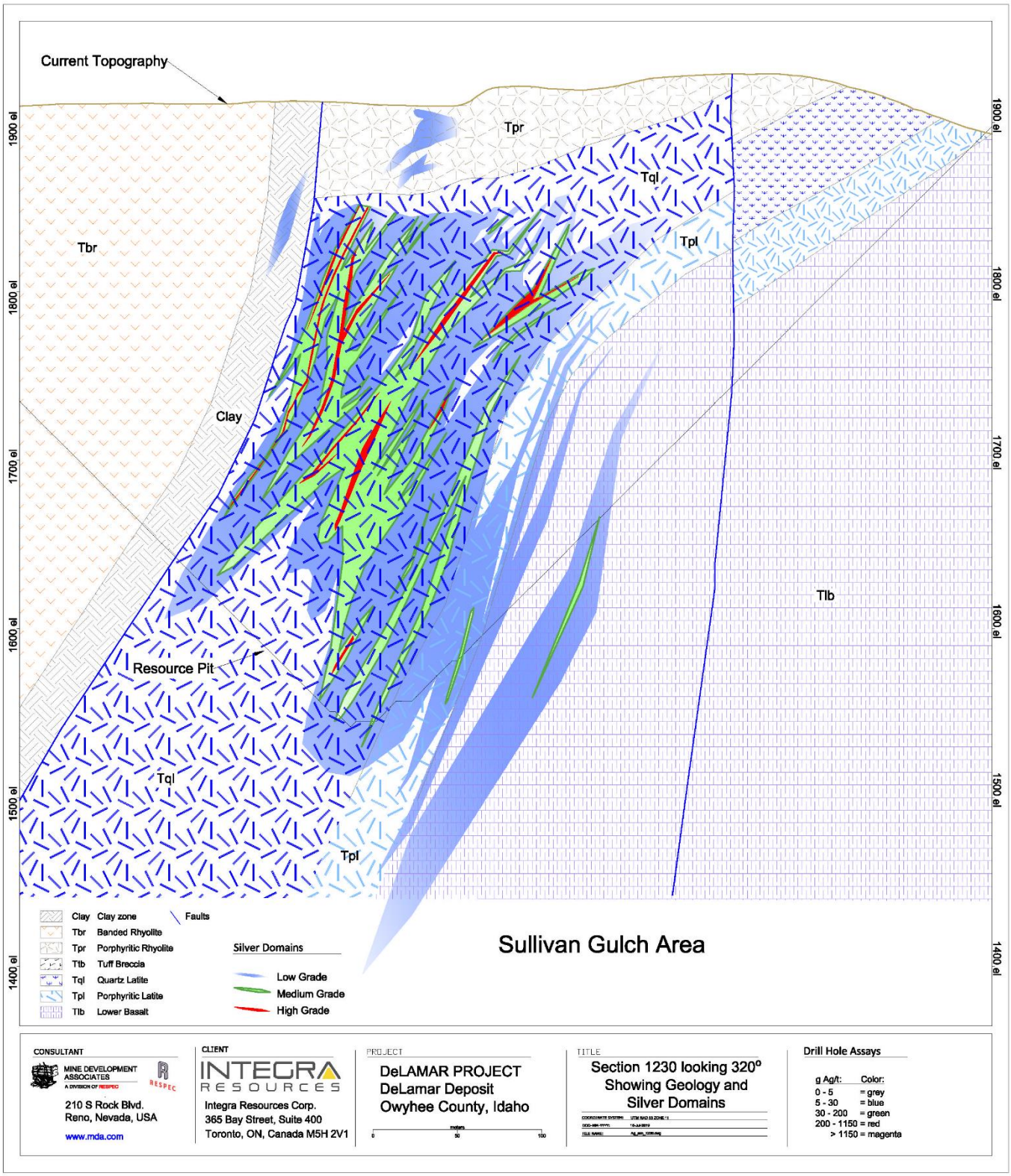


Figure 14.3 Cross Section 2010 NW Showing Sommercamp and N. DeLamar Gold Domains

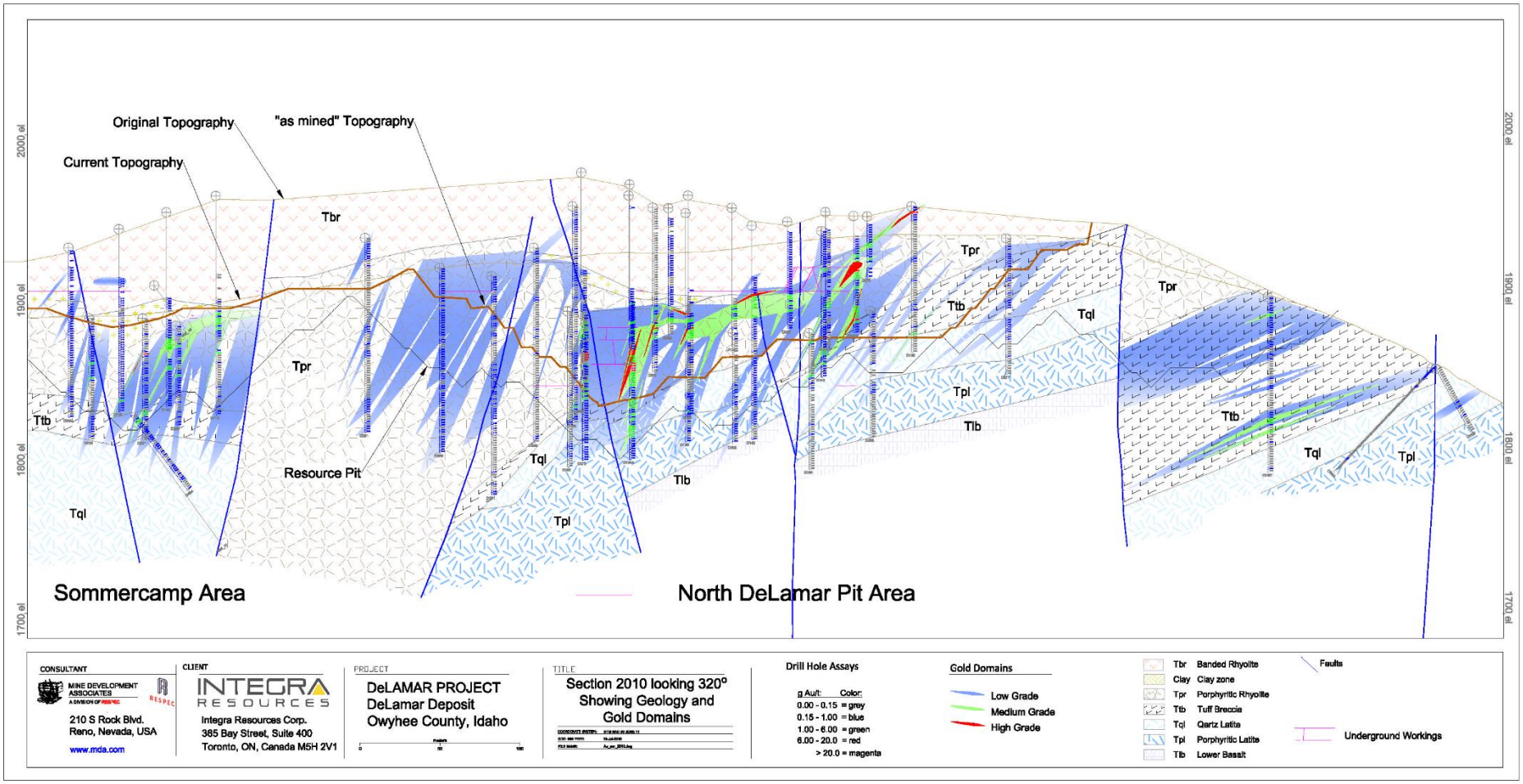


Figure 14.4 Cross Section 2010 NW Showing Sommercamp and N. DeLamar Silver Domains

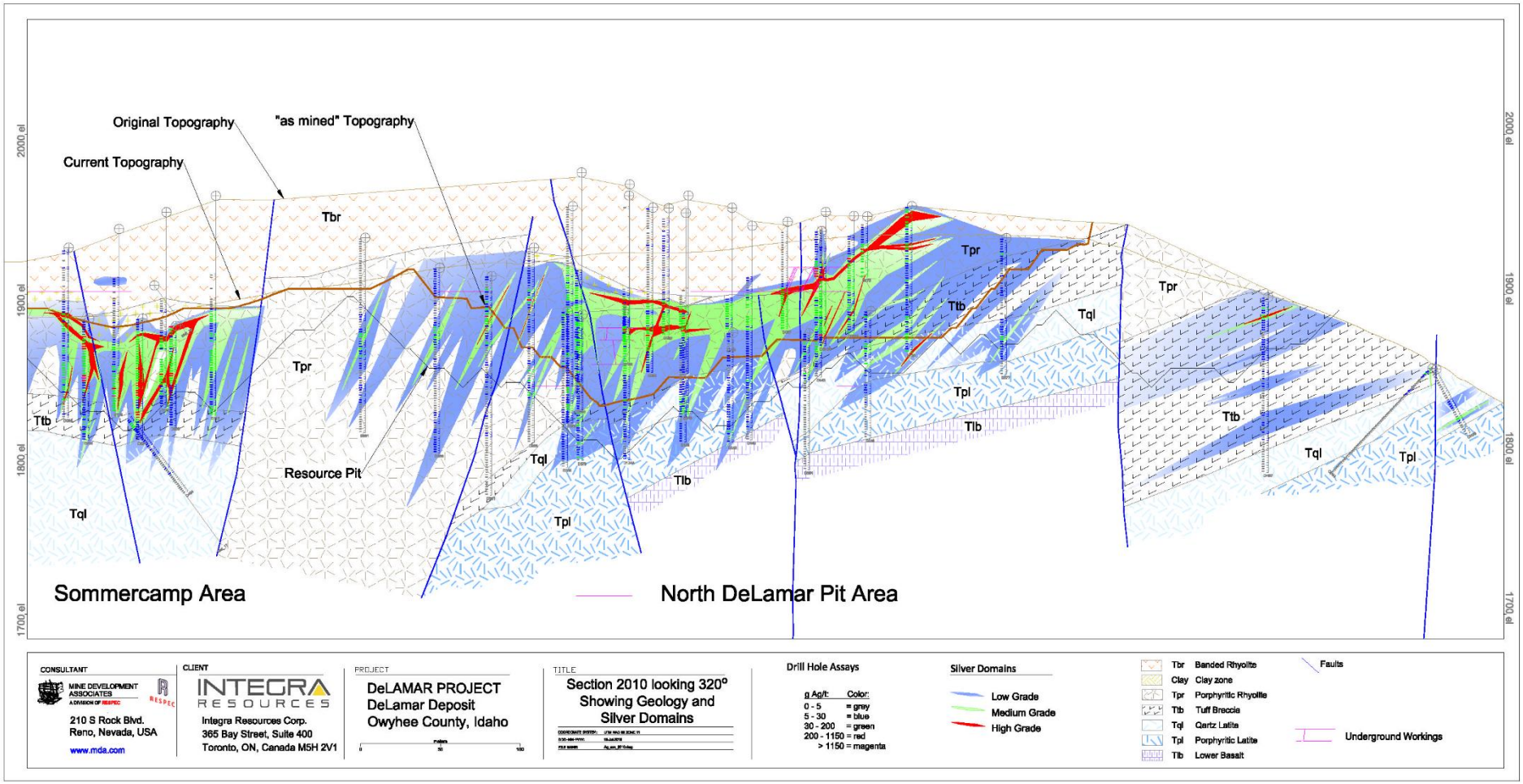


Figure 14.5 Cross Section 2790 NW Showing Gold Domains at Glen Silver

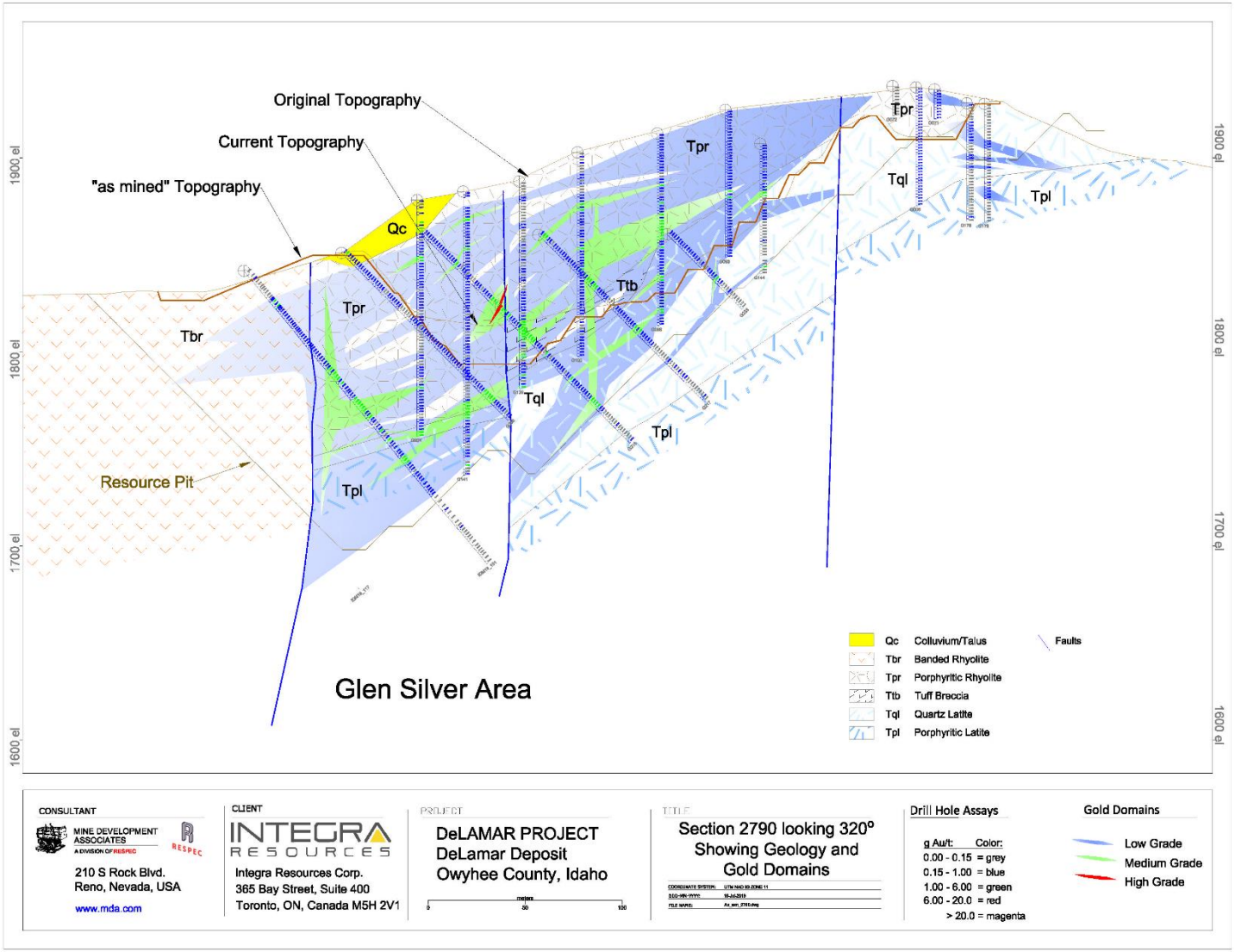
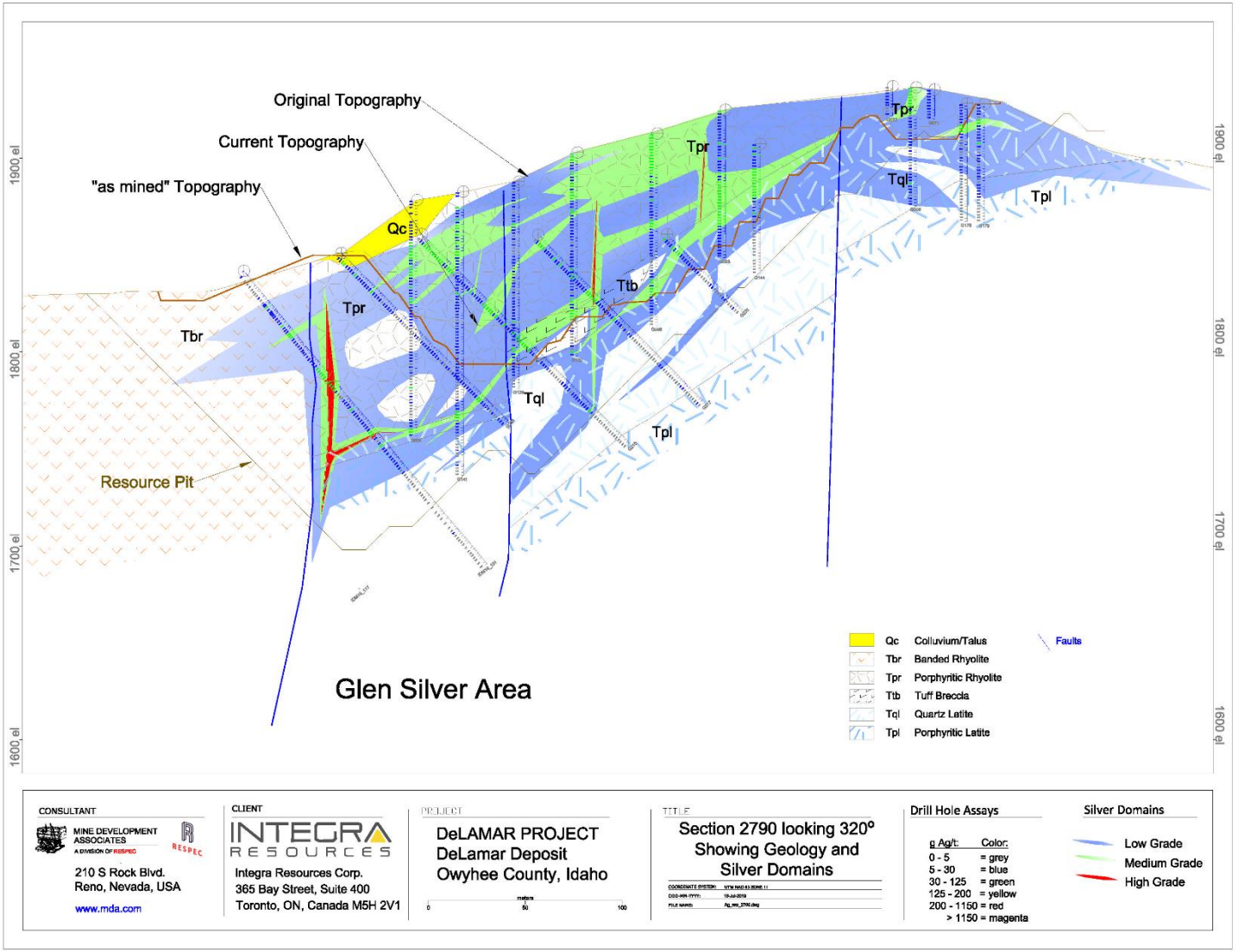


Figure 14.6 Cross Section 2790 NW Showing Silver Domains at Glen Silver



The final cross-sectional gold and silver mineral-domain polygons were projected three-dimensionally to the drill data in each sectional window, and these three-dimensional polygons were then sliced horizontally at six-meter elevation intervals that match the mid-block elevations of the resource block model. The slices were used in the southern portion of the deposit, including the Sommercamp, North DeLamar, and Sullivan Gulch areas, to create a new set of horizontal mineral-domain polygons for both gold and silver on level plans at six-meter spacings. Level plans were used in this portion of the DeLamar deposit due to the predominance of steeply dipping mineralization, especially in medium- and high-grade domains.

For the North and South Wahl, Glen Silver, and Milestone areas, where relatively shallowly dipping mineralization predominates, the three-dimensional cross-sectional gold and silver domain polygons were sliced vertically at six-meter intervals, and these slices were used to interpret the domains on long sections.

Wireframe surfaces of faults, high-angle mineralized structures, and important lithologic contacts that focus or terminate mineralization were used to assist in the rectification of the mineral domains on long sections and level plans. The completed level-plan and long-section mineral-domain polygons serve to rectify the gold and silver domains to the drill-hole data at the scale of the block model.

14.8.2 Assay Coding, Capping, and Compositing

Drill-hole gold and silver assays were coded to the gold and silver mineral domains, respectively, using their respective cross-sectional polygons. Assay caps were determined by the inspection of population distribution plots of the coded assays grouped by domain to identify high-grade outliers that might be appropriate for capping. The plots were also evaluated for the possible presence of multiple grade populations within any of the domains. Descriptive statistics of the coded assays by domain and visual reviews of the spatial relationships of the possible outliers, and their potential impacts during grade interpolation, were also considered in the definition of the assay caps (shown in Table 14.4).

Each model block was coded to the volume percentage of each of the three modeled domains for both gold and silver, as discussed below. Volumes of blocks that were not entirely coded to the lower- and higher-grade mineral domains for either or both metals were assigned to domain "0" and were estimated using assays lying outside of the modeled domains (uncoded assays). The uncoded assays used in this dilutionary estimate were also coded, as shown in Table 14.4.

Table 14.4 DeLamar Area Gold and Silver Assay Caps by Domain

Domain	g Au/t	No. of Samples Capped (% of samples)	g Ag/t	No. of Samples Capped (% of samples)
0	2	47 (<1%)	160	17 (<1%)
100	3	32 (<1%)	125	19 (<1%)
200	6	20 (<1%)	200	41 (<1%)
300	40	5 (1.5%)	1,325	27 (1.3%)

Descriptive statistics of the capped and uncapped coded assays are provided in Table 14.5 and Table 14.6 for gold and silver, respectively.

Table 14.5 Descriptive Statistics of DeLamar Area Coded Gold Assays

Domain	Assays	Count	Mean (g Au/t)	Median (g Au/t)	Std. Dev.	CV	Min. (g Au/t)	Max. (g Au/t)
0	Au	48,118	0.10	0.07	0.78	8.02	0.00	102.86
	Au Cap	48,118	0.09	0.07	0.13	1.46	0.00	2.00
100	Au	38,401	0.37	0.34	0.26	0.69	0.00	9.70
	Au Cap	38,401	0.37	0.34	0.24	0.64	0.00	3.00
200	Au	5,033	1.73	1.37	1.60	0.93	0.00	61.03
	Au Cap	5,033	1.69	1.37	1.09	0.64	0.00	6.00
300	Au	327	13.72	8.95	23.82	1.74	0.17	368.64
	Au Cap	327	11.93	8.95	7.95	0.67	0.17	40.00
100+200+300	*Au*	*43,761*	*0.63*	*0.34*	*2.45*	*3.91*	*0.00*	*368.64*
	Au Cap	*43,761*	*0.61*	*0.34*	*1.34*	*2.19*	*0.00*	*40.00*

Table 14.6 Descriptive Statistics of DeLamar Area Coded Silver Assays

Domain	Assays	Count	Mean (g Ag/t)	Median (g Ag/t)	Std. Dev.	CV	Min. (g Ag/t)	Max. (g Ag/t)
0	Ag	28,179	3.3	1.0	12.9	3.9	0.0	791.0
	Ag Cap	28,179	3.2	1.0	9.5	3.0	0.0	160.0
100	Ag	26,524	13.0	10.3	10.2	0.8	0.0	186.3
	Ag Cap	26,524	13.0	10.3	10.1	0.8	0.0	125.0
200	Ag	12,703	56.3	47.0	35.2	0.6	0.0	732.9
	Ag Cap	12,703	56.0	47.0	33.1	0.6	0.0	200.0
300	Ag	298	297.7	205.2	371.7	1.3	6.9	7877.3
	Ag Cap	280	280.3	205.2	227.0	0.8	6.9	1325.0
100+200+300	*Ag*	*41,178*	*39.5*	*17.1*	*102.1*	*2.6*	*0.0*	*7877.3*
	Ag Cap	*41,178*	*38.6*	*17.1*	*77.3*	*2.0*	*0.0*	*1325.0*

In addition to the assay caps, restrictions on the search distances of higher-grade composites of some of the domains were applied during grade interpolations (discussed further below). Search restrictions can minimize the number of samples subjected to capping while properly respecting the highest-grade populations within each domain.

The capped assays were composited at 3.05 meter (10-foot) down-hole intervals respecting the mineral domains. Descriptive statistics of DeLamar composites are shown in Table 14.7 and Table 14.8 for gold and silver, respectively.

Table 14.7 Descriptive Statistics of DeLamar Area Gold Composites

Domain	Count	Mean (g Au/t)	Median (g Au/t)	Std. Dev.	CV	Min. (g Au/t)	Max. (g Au/t)
0	28,628	0.09	0.07	0.12	1.32	0.00	2.00
100	21,393	0.37	0.34	0.20	0.54	0.00	3.00
200	3,264	1.69	1.39	0.94	0.55	0.00	6.00
300	244	11.93	9.43	7.16	0.60	0.21	40.00
100+200+300	*24,901*	*0.61*	*0.34*	*1.28*	*2.11*	*0.00*	*40.00*

Table 14.8 Descriptive Statistics of DeLamar Area Silver Composites

Domain	Count	Mean (g Ag/t)	Median (g Ag/t)	Std. Dev.	CV	Min. (g Ag/t)	Max. (g Ag/t)
0	16,562	3.2	1.4	8.9	2.8	0.0	160.0
100	15,121	13.0	11.1	8.6	0.7	0.0	125.0
200	7,562	56.0	48.5	28.4	0.5	0.0	200.0
300	1,264	280.3	215.0	202.5	0.7	6.9	1325.0
100+200+300	*23,947*	*38.6*	*17.1*	*73.4*	*1.9*	*0.0*	*1325.0*

14.8.3 Block Model Coding

The 6.0-meter-spaced level-plan and long-sectional mineral-domain polygons were used to code 6.0 x 6.0 x 6.0-meter blocks with a model bearing of 320°. The percentage volume of each mineral domain, as coded directly by the level plans and long sections, is stored within each block, as is the volume percentage of the block that lies outside of the modeled gold and silver domains (the "partial percentages").

Two topographic surfaces were used to code the block model: the as-mined and present-day surfaces discussed in Section 14.2.2. These digital topographic surfaces were used to define: (1) the percentage of each block that lies within bedrock; and (2) the percentage of each block that is comprised of backfill/dump material, which lies above the as-mined surface and below the present-day surface.

The modeled mineralization has a variety of orientations, which led to the construction of wireframe solids to encompass model areas with unique orientations of the mineralization. These solids were then used to code the model blocks to these specific areas.

The oxidation wire-frame solids described in Section 14.6 were used to code model blocks as oxidized, transitional, or unoxidized.

Finally, the specific-gravity values discussed in Section 14.7 were assigned to model blocks coded as bedrock, according to oxidation state, or backfill/dump. The specific-gravity values were then used in combination with the percentages of rock and fill for each block to determine the tonnage of the block.

14.8.4 Grade Interpolation

Parameters used in the estimation of gold and silver grades are summarized in Table 14.9.

Table 14.9 Summary of DeLamar Area Grade Estimation Parameters

Estimation Pass – Au + Ag Domain	Search Ranges (meters)			Composite Constraints		
	Major	Semi-Major	Minor	Min	Max	Max/Hole
Pass 1 + 2 - Doman 100	60	60	30	2	12	4
Pass 1 + 2 - Doman 200 + 300 + 0	60	60	30	2	20	4
Pass 3 - Doman 0 + 100 + 200 +300	170	170	170	1	20	4

Restrictions on Search Ranges

Domain	Search Restriction Threshold	Search Restriction Distance	Estimation Pass
Au 100	>0.7 g Au/t	40 meters	1, 2
Au 300	>20 g Au/t	35 meters	1, 2, 3
Ag 300	>400 g Ag/t	35 meters	1, 2, 3
Au 0	>0.5 g Au/t	6 meters	1, 2, 3
Ag 0	>34.29 g Ag/t	9 meters	1, 2, 3

Statistical analyses of coded assays and composites, including coefficients of variation and population-distribution plots, indicate multiple populations of significance were captured in the higher-grade domain (domain 300) of both gold and silver, as well as in the low-grade gold domain (domain 100). The recognition of multiple populations within these domains, coupled with the results of initial grade-estimation runs in which higher-grade samples in these multi-population domains were affecting inappropriate volumes in the block model, led to the use of restrictions on the search distances for the higher-grade populations of these domains. The search restrictions place limits on the maximum distances from a block that the high-grade population composites can be 'found' and used in the interpolation of gold and/or silver grade into that block. The final search-restriction parameters were derived from the results of multiple interpolation iterations that employed various search-restriction distances. Severe search restrictions were used for the gold and silver estimated in domain 0, as domain 0 composites of any substantive grade involve assay data that are not modeled within the mineral domains due to the lack of continuity and/or lack of geologic context.

The maximum number of composites allowed for the estimation of the low-grade domains of gold and silver in Pass 1 and Pass 2 are less than that of the other grade interpolations. This was done to decrease the smearing of outlier high grades that are present in this otherwise low-grade domain.

The gold and silver mineralization commonly exhibits multiple orientations in any single area. Most commonly, high-angle structurally controlled mineralization is often associated with related lower-angle mineralization that spreads outwards from the high-angle mineralization. A total of 13 unique dips were identified in the DeLamar resource area. Two dominant strike directions were also identified, a 320° strike direction for most of the model area and 340° for a portion of the Sommercamp area. The 13 dips and two strikes combine to create 18 unique orientation areas distributed throughout the model area, and each area was coded into the block model using wireframed 'estimation area' solids.

Over half of the estimation areas are characterized a single strike but two dips, which are accounted for in grade interpolation by the use of two initial passes, Pass 1 and Pass 2. The dip that reflects higher-grade mineralization was given priority, which most commonly meant the steeper of the two dips. The priority dip was then used in the search ellipse for the Pass 1 grade interpolation, while Pass 2 estimation used the secondary dip in its search ellipse. All other estimation parameters, such as search distance and sample criteria, remained identical in the two passes (Table 14.9). The third and final estimation pass was an isotropic pass, *i.e.* without an orientation bias, and was used to interpolate grades that were not estimated in the first two passes.

Gold and silver grades were interpolated using inverse-distance to the third power, ordinary-krige, and nearest-neighbor methods. The mineral resources reported herein were estimated by the inverse-distance interpolation, as this method led to results that were judged to more closely approximate the drill data than those obtained by ordinary kriging. The nearest-neighbor estimation was completed as a check on the inverse-distance and krige interpolations.

Grade interpolation was completed using length-weighted 3.05-meter (10-foot) composites. The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded to that domain. Blocks coded as having partial percentages of more than one gold and/or silver domain had multiple grade interpolations, one for each domain coded into the block for each metal. The estimated grades for each gold and silver domain coded to a block were coupled with the partial percentages of the those mineral domains in the block, as well as any outside, dilutionary, domain 0 grades and volumes, to enable the calculation of a single volume-averaged gold and a single volume-averaged silver grade for each block. These single final resource block grades, and their associated resource tonnages, are therefore fully block-diluted using this methodology.

14.8.5 Model Checks

Polygonal sectional volumes derived from the sectional mineral-domain polygons were compared to the polygonal volumes derived from the level plans and long sections, as well as to the coded block-model volumes derived from the partial percentages, to assure close agreement. All block-model coding, including topographies, oxidation, estimation areas, and mineral domains, was checked visually on the computer. A polygonal grade and tonnage estimate using the cross-sectional domain polygons and the nearest-neighbor and ordinary-krige estimates were all used as a check on the inverse-distance estimation results. No unexpected relationships between the check estimates and the inverse-distance estimate were identified. Various grade-distribution plots of assays and composites and nearest-neighbor, ordinary-krige, and inverse-distance block grades were evaluated as a check on both the global and local estimation results. Finally, the inverse-distance grades were visually compared to the drill-hole assay data to assure that reasonable results were obtained.

14.9 Florida Mountain Area Gold and Silver Modeling

The modeling procedures employed for the Florida Mountain resources were very similar to those used in the estimation of the DeLamar area resources (Section 14.8). The following summary of the Florida Mountain resource modeling is therefore discussed in less detail.

14.9.1 Mineral Domains

The approximate grade ranges of the low-grade (domain 100), mid-grade (domain 200), and high-grade (domain 300) grade populations and mineral domains at Florida Mountain are listed in Table 14.10.

Table 14.10 Approximate Grade Ranges of Florida Mountain Area Gold and Silver Domains

Domain	g Au/t	g Ag/t
100	~0.2 to ~0.6	~7 to ~30
200	~0.6 to ~2.0	~30 to ~90
300	> ~2.0	> ~90

The Florida Mountain gold and silver mineralization was modeled by interpreting gold and silver mineral-domain polygons separately on a set of vertical, 30-meter spaced, north-looking, east-west cross sections that span the presently known extents of the deposit. The mineral domains were interpreted using the gold and silver drill-hole assay data, associated drill-hole lithologic codes, documented descriptions of the mineralization, Integra's cross-sectional lithologic modeling, and wireframe solids of the historical underground workings created by MDA.

At Florida Mountain, a series of thin, anastomosing, and steeply dipping veins and breccias characterize the mineralization. These thin veins and breccias are enveloped by mineralization modeled in the low-grade gold and silver domains, and they are cored by mid- and high-grade domain mineralization. The continuity of any single vein or vein-breccia decreases as the grade increases, although zones characterized by these intermittent higher-grade domains do have strike continuity, and many of these zones correlate with historically named vein zones.

The mineral-domain modeling relied largely on grade values, because the project digital database lacks information on quartz veins, alteration (especially silicification), etc. In addition, the domain modeling was also guided by Integra's lithologic sections, historical descriptions of mineralized orientations, and the wireframes of the underground workings constructed by MDA. The high density of drill data at Florida Mountain ultimately overcame any limits imparted by the somewhat limited geologic inputs.

The Florida Mountain mineralization was modeled over a northly strike extent of almost 1,400 meters, an east-west width of up to 675 meters, and an elevation range of 465 meters. Cross-sections showing examples of the gold and silver mineral domains for the Florida Mountain deposit are shown in Figure 14.7 through Figure 14.10.

The final cross-sectional gold and silver mineral-domain polygons were projected three-dimensionally to the drill data in each sectional window, and these three-dimensional polygons were then sliced horizontally at 6.0-meter elevation intervals that match the mid-block elevations of the resource block model. The horizontal slices were used to create a new set of mineral-domain polygons for both gold and silver on level plans at 6.0-meter spacings that serve to rectify the domain interpretations to the drill-hole data at the scale of the block model. Level plans were used due to the steeply dipping mineralization that characterizes the entire Florida Mountain deposit.

14.9.2 Assay Coding, Capping, and Compositing

Drill-hole gold and silver assays were coded to the Florida Mountain gold and silver mineral domains using their respective cross-sectional polygons, and assay caps were defined for each domain, as well as for drill-hole assays that lie outside of the modeled domains (assigned to domain "0") as summarized in Table 14.11. In addition to the assay caps, restrictions on the search distances of higher-grade portions of the domains were applied during grade interpolations (discussed further below).

Table 14.11 Florida Mountain Area Gold and Silver Assay Caps by Domain

Domain	g Au/t	Number Capped (% of samples)	g Ag/t	Number Capped (% of samples)
0	5.0	51 (<1%)	100	14 (<1%)
100	3.0	11 (<1%)	65	35 (<1%)
200	9.0	1 (<1%)	n/a	n/a
300	75.0	12 (<1%)	900	19 (3%)

Figure 14.7 Florida Mountain Cross Section 2830 N Showing Geology and Gold Domains

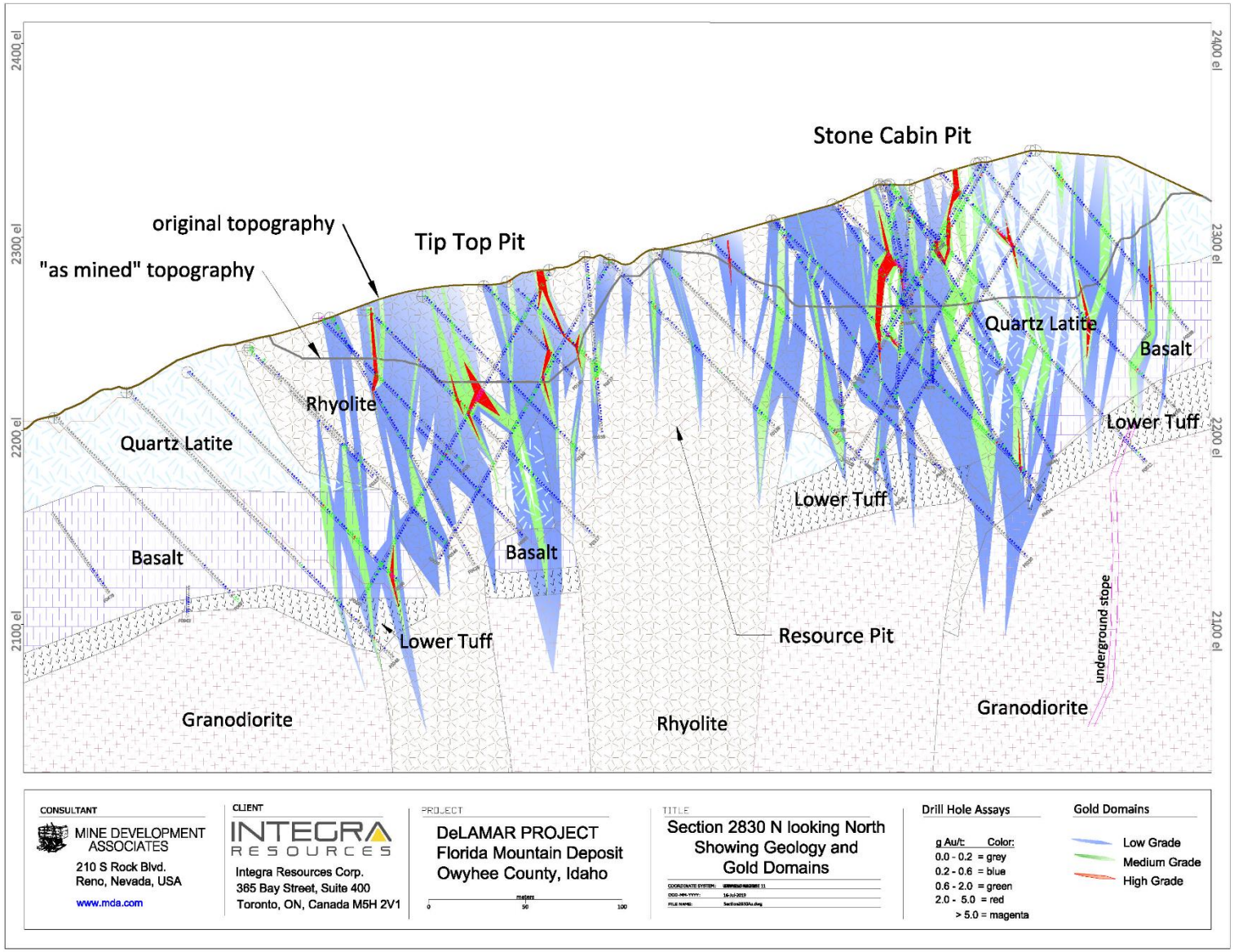


Figure 14.8 Florida Mountain Cross Section 2830 N Showing Geology and Silver Domains

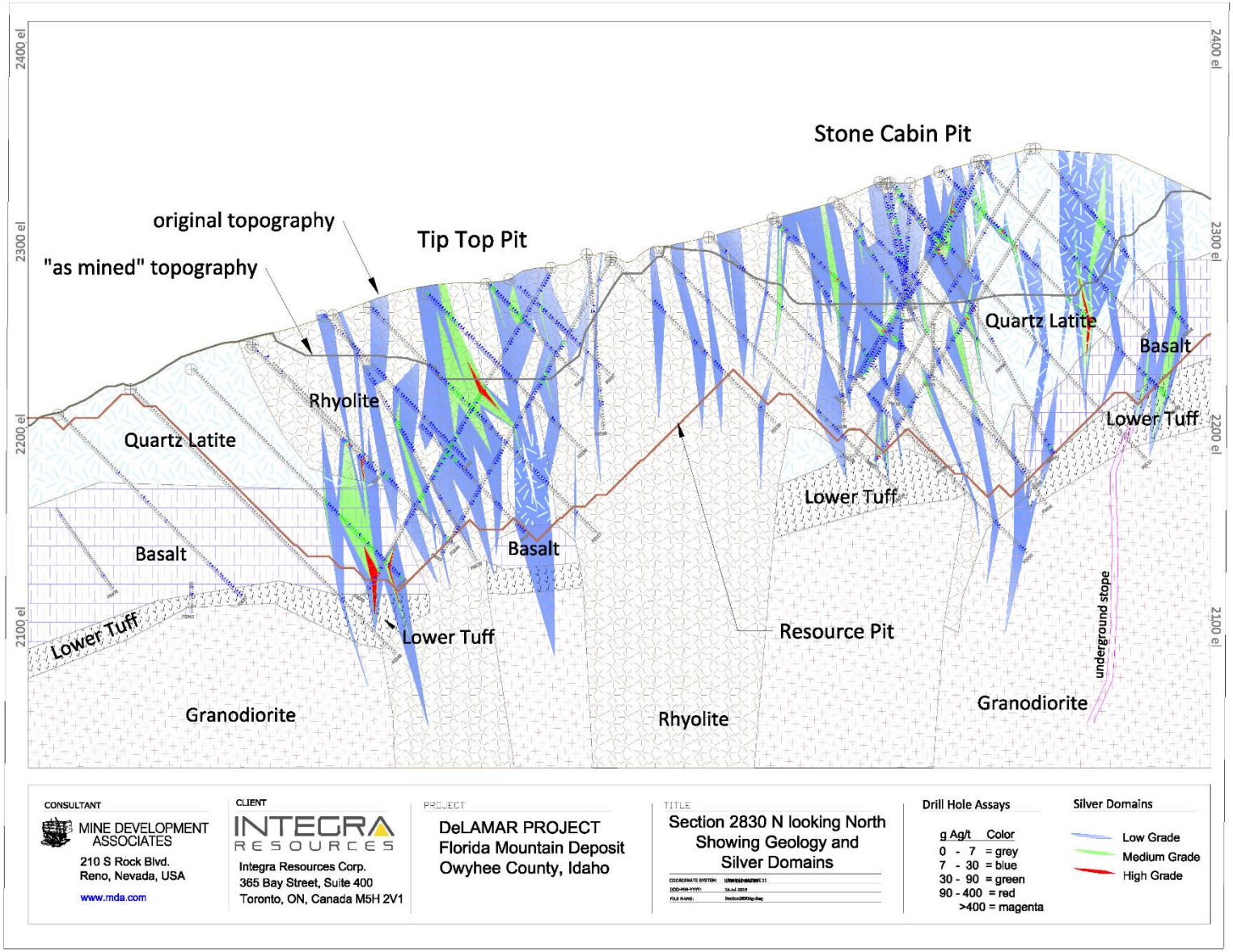


Figure 14.9 Florida Mountain Cross Section 3280 N Showing Geology and Gold Domains

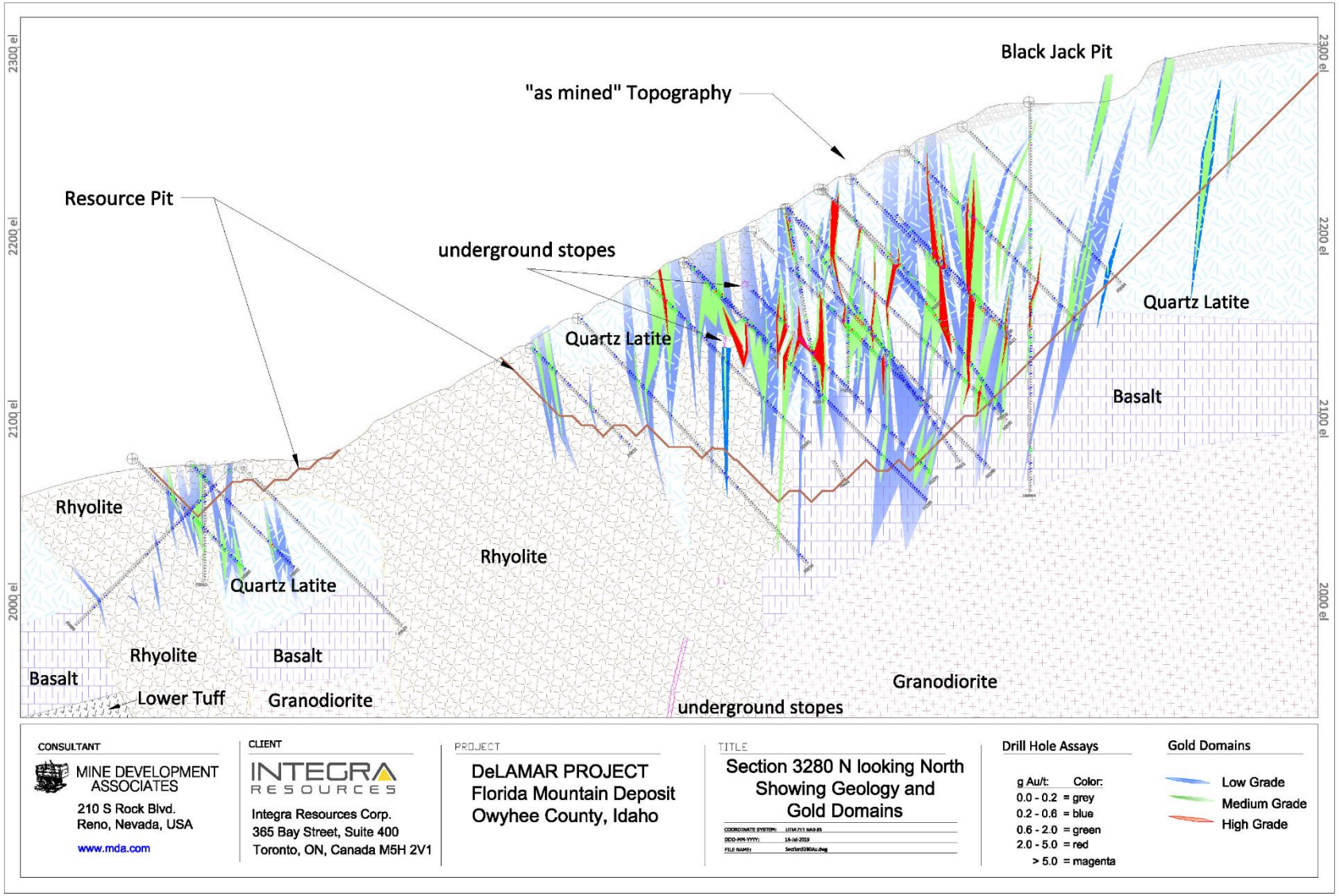
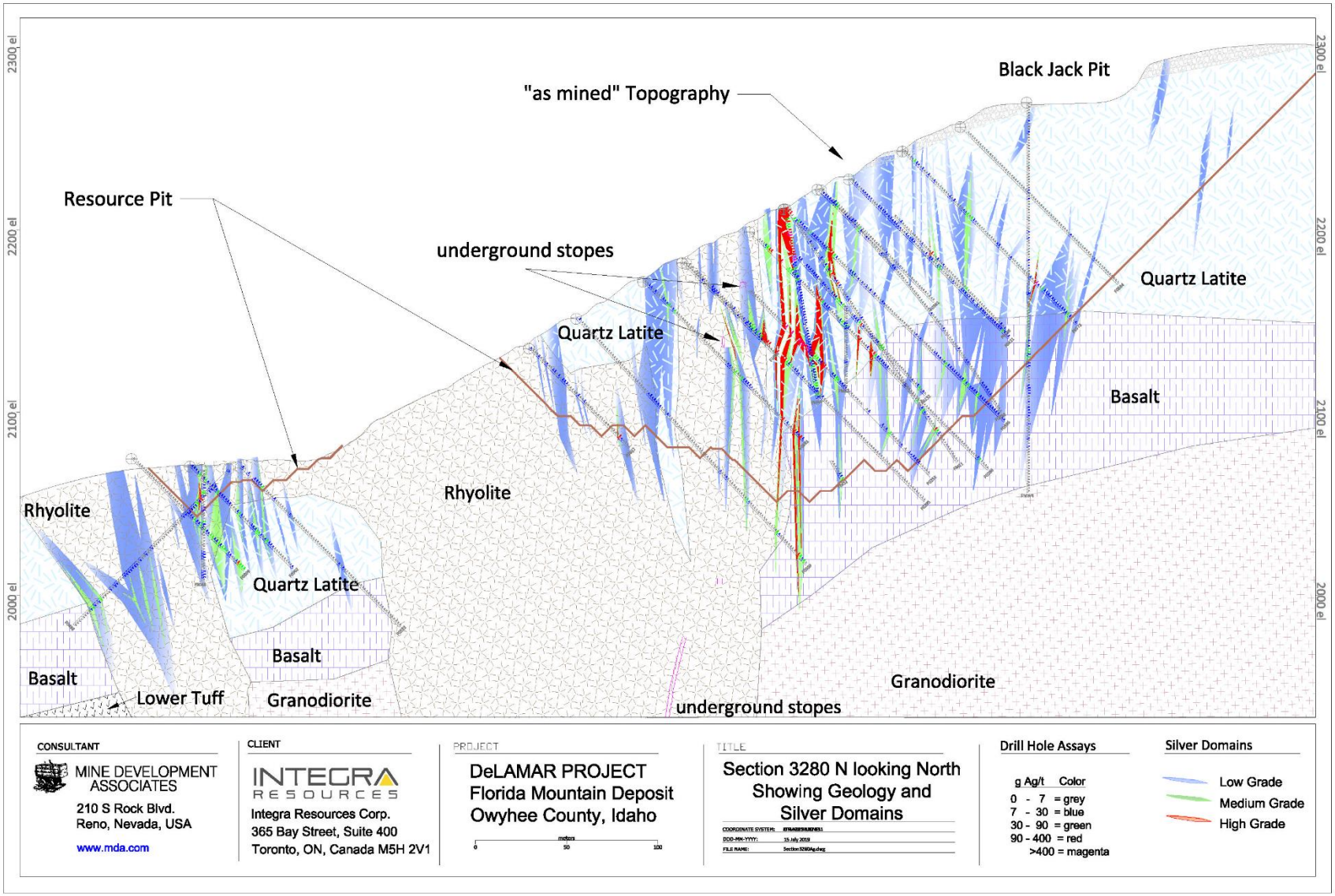


Figure 14.10 Florida Mountain Cross Section 3280 N Showing Geology and Silver Domains



Descriptive statistics of the uncapped and capped coded assays are provided in Table 14.12 and Table 14.13 for gold and silver, respectively.

Table 14.12 Descriptive Statistics of Florida Mountain Area Coded Gold Assays

Domain	Assays	Count	Mean (g Au/t)	Median (g Au/t)	Std. Dev.	CV	Min. (g Au/t)	Max. (g Au/t)
0	Au	46,846	0.13	0.07	0.53	4.15	0.00	43.99
	Au Cap	46,846	0.12	0.07	0.27	2.25	0.00	5.00
100	Au	20,957	0.34	0.27	0.23	0.68	0.00	3.63
	Au Cap	20,957	0.34	0.27	0.23	0.67	0.00	3.00
200	Au	7,174	1.06	0.86	0.78	0.74	0.00	12.31
	Au Cap	7,174	1.06	0.86	0.78	0.74	0.00	9.00
300	Au	1,373	7.87	3.70	16.60	2.11	0.03	286.22
	Au Cap	1,373	7.24	3.70	10.68	1.48	0.03	75.00
100+200+300	*Au*	*29,504*	*0.87*	*0.38*	*3.96*	*4.57*	*0.00*	*286.22*
	Au Cap	*29,504*	*0.84*	*0.38*	*2.77*	*3.31*	*0.00*	*75.00*

Table 14.13 Descriptive Statistics of Florida Mountain Area Coded Silver Assays

Domain	Assays	Count	Mean (g Ag/t)	Median (g Ag/t)	Std. Dev.	CV	Min. (g Ag/t)	Max. (g Ag/t)
0	Ag	36,842	2.5	1.7	11.6	4.7	0.0	1865.6
	Ag Cap	36,842	2.4	1.7	3.9	1.6	0.0	100.0
100	Ag	15,844	12.0	9.9	7.7	0.6	0.0	90.9
	Ag Cap	15,844	11.9	9.9	7.5	0.6	0.0	65.0
200	Ag	3,193	46.1	39.8	25.4	0.6	0.7	293.5
	Ag Cap	3,193	46.1	39.8	25.4	0.6	0.7	293.5
300	Ag	245	245.2	143.3	391.7	1.6	7.9	6057.3
	Ag Cap	219	219.3	143.3	188.3	0.9	7.9	900.0
100+200+300	*Ag*	*19,676*	*25.1*	*12.0*	*83.0*	*3.3*	*0.0*	*6057.3*
	Ag Cap	*19,676*	*24.3*	*12.0*	*52.2*	*2.2*	*0.0*	*900.0*

The capped assays were composited at 3.05 meter (10-foot) down-hole intervals respecting the mineral domains. Descriptive statistics of Florida Mountain composites are shown in Table 14.14 and Table 14.15 for gold and silver, respectively.

Table 14.14 Descriptive Statistics of Florida Mountain Area Gold Composites

Domain	Count	Mean (g Au/t)	Median (g Au/t)	Std. Dev.	CV	Min. (g Au/t)	Max. (g Au/t)
0	25,295	0.12	0.09	0.24	2.04	0.00	5.00
100	12,191	0.34	0.31	0.18	0.53	0.00	2.50
200	4,510	1.06	0.91	0.66	0.63	0.00	8.37
300	975	7.24	4.15	8.82	1.22	0.03	69.39
100+200+300	*17,676*	*0.84*	*0.38*	*2.43*	*2.91*	*0.00*	*69.39*

Table 14.15 Descriptive Statistics of Florida Mountain Area Silver Composites

Domain	Count	Mean (g Ag/t)	Median (g Ag/t)	Std. Dev.	CV	Min. (g Ag/t)	Max. (g Ag/t)
0	20,165	2.4	1.7	3.4	1.4	0.0	100.0
100	9,473	11.9	10.3	6.5	0.5	0.0	65.0
200	2,141	46.1	40.8	21.7	0.5	1.2	243.1
300	449	219.3	148.1	173.8	0.8	7.9	900.0
100+200+300	*12,063*	*24.3*	*12.2*	*50.2*	*2.1*	*0.0*	*900.0*

14.9.3 Block Model Coding

The 6.0-meter-spaced level-plan mineral-domain polygons were used to code a block model with a model bearing of 000° and blocks that are 6-meter cubes. The percentage volume of each mineral domain, as well as the percentage of any volume in the block lying outside the mineral domains, is stored within each block (the "partial percentages").

Two topographic surfaces were used to code the block model: the as-mined and present-day surfaces discussed in Section 14.2.2. These digital topographic surfaces were used to define: (1) the percentage of each block that lies within bedrock; and (2) the percentage of each block that is comprised of backfill/dump material, which lies above the as-mined surface and below the present-day surface.

The modeled mineralization has a variety of orientations, which led to the construction of wireframe solids to encompass model areas with unique orientations. These solids were then used to code the model blocks to these specific areas.

The oxidation solids described in Section 14.6 were used to code model blocks as oxidized, transitional, or unoxidized. The partial percentages of the wireframe solids of the historical underground workings (see Section 14.2.3) were also coded into model blocks.

Finally, the specific-gravity values discussed in Section 14.7 were assigned to model blocks coded as bedrock, according to its oxidation state, or backfill/dump. The specific-gravity values were then used in combination with the percentages of rock and fill for each block to determine the tonnage of the block.

14.9.4 Grade Interpolation

Multiple populations of significance were captured in the high-grade domain (domain 300) of both gold and silver, which led to the incorporation of search restrictions. Search restrictions were also used for the dilutionary material outside the mineral domains (domain 0) for both the gold and silver grade estimations.

The maximum number of composites allowed for the estimation of the low-grade domains of gold and silver in Passes 1 and 2 are less than that for all other grade interpolations. This was done to decrease the smearing of outlier grades that occur in this otherwise low-grade domain.

Gold and silver grades were interpolated using inverse distance to the third power, ordinary krige, and nearest-neighbor methods. The mineral resources reported herein were estimated by the inverse-distance interpolation, as this method led to results that were judged to more closely approximate the drill data than those obtained by ordinary kriging. The nearest-neighbor estimation was completed as a check on the inverse-distance and krige interpolations. The parameters applied to the gold-grade estimations at Florida Mountain are summarized in Table 14.16.

Table 14.16 Summary of Florida Mountain Area Estimation Parameters

Estimation Pass – Au + Ag Domain	Search Ranges (meters)			Composite Constraints		
	Major	Semi-Major	Minor	Min	Max	Max/Hole
Pass 1 - Domain 100	60	60	30	2	12	4
Pass 1 - Domain 200 + 300 + 0	60	60	30	2	20	4
Pass 2 - Domain 100	120	120	120	1	20	4
Pass 2 - Domain 200 + 300 + 0	120	120	120	1	12	4

Restrictions on Search Ranges

Domain	Search Restriction Threshold	Search Restriction Distance	Estimation Pass
Au 300	>10 g Au/t	20 meters	1, 2
Ag 300	>400 g Ag/t	35 meters	1, 2
Au 0	>1.0 g Au/t	6 meters	1, 2
Ag 0	>6.0 g Ag/t	6 meters	1, 2

Estimation areas were defined for the purposes of the Florida Mountain grade interpolations, each characterized by a single dominant orientation of mineralization. Three strike directions (332°, 342°, and 000°) and four dips (-75° to vertical, dipping to the east and west) combine to define the five estimation areas that were constructed.

Grade interpolation was completed in two passes using length-weighted 3.05-meter (10-foot) composites. The second pass was used to estimate grades into blocks that were not estimated in Pass 1. The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded by that domain. The estimated grades for each gold and silver domain coded to a block were coupled with the partial percentages of the those mineral domains in the block, as well as the outside, dilutionary, domain 0 grades and volumes, to enable the calculation of a single volume-averaged gold and a single volume-averaged silver grade for each block. These single resource block grades, and their associated resource tonnages, are therefore fully block-diluted using this methodology.

14.9.5 Model Checks

The model and estimation were checked in a similar manner as described for the DeLamar deposit estimation in Section 14.8.5.

14.10 DeLamar Project Mineral Resources

The DeLamar project mineral resources have been estimated to reflect potential open-pit extraction and potential processing by a combination of heap leaching, milling / agitated leaching, and flotation. To meet the requirement of the in-pit resources having reasonable prospects for eventual economic extraction, pit optimizations for the DeLamar and Florida Mountain areas were run using the parameters summarized in Table 14.17 and Table 14.18.

Table 14.17 Pit Optimization Cost Parameters

Parameter	DeLamar	Florida Mtn	Unit
Mining Cost	$ 2.20	$ 2.20	$/tonne mined
Heap Leach Processing	$ 3.35	$ 3.35	$/tonne processed
Mill / Agitated Leach Processing	$	$ 10.00	$/tonne processed
Flotation Processing	$ 12.00	$	$/tonne processed
G&A Cost	$ 4,000	$ 4,000	$1,000s/year
Tonnes per Day	15,000	15,000	tonnes-per-day processed
Tonnes per Year	5,250	5,250	1000s tonnes-per-year processed
G&A per Ton	$ 0.76	$ 0.76	$/tonne processed
Au Price	$ 1,400	$ 1,400	$/oz produced
Ag Price	$ 18	$ 18	$/oz produced
Au Refining Cost	$ 5.00	$ 5.00	$/oz produced
Ag Refining Cost	$ 0.50	$ 0.50	$/oz produced
NSR Royalty	1%	0%	

Table 14.18 Pit-Optimization Metal Recoveries by Deposit and Oxidation State

	DeLamar			Florida Mountain		
Process Type	Oxidized	Transitional	Unoxidized	Oxidized	Transitional	Unoxidized
Leach Recovery - Au	85%	80%	-	85%	80%	-
Leach Recovery - Ag	45%	40%	-	45%	40%	-
Mill/Leach Recovery - Au	-	-	-	-	-	86%
Mill/Leach Recovery - Ag	-	-	-	-	-	63%
Flotation Recovery - Au	-	-	90%	-	-	-
Flotation Recovery - Ag	-	-	95%	-	-	-

The pit shells created using these optimization parameters were applied to constrain the project resources for both the DeLamar and Florida Mountain deposits. The in-pit resources were further constrained by the application of a gold-equivalent cutoff of 0.2 g/t to all model blocks lying within the optimized pits that are coded as oxidized or transitional, and 0.3 g/t for blocks coded as unoxidized. Gold equivalency, as used in the application of the resource cutoffs, is a function of metal prices (Table 14.17) and metal recoveries, with the recoveries varying by deposit and oxidation state (Table 14.18). These variables, combined with the estimated gold and silver grades, are used to calculate a gold-

equivalent grade for every block in the model. An example of the calculation of the gold-equivalent grade ("g AuEq/t") of an unoxidized block from the Florida Mountain resource model is as follows:

$$g\ AuEq/t = g\ Au/t + (g\ Ag/t \div ((1{,}400 \times 0.86) \div (18 \times 0.63))$$

where "g Au/t" and "g Ag/t" are the estimated gold and silver block-diluted grades, respectively, and the other parameters are the metal prices and recoveries. The gold-equivalent grades are calculated for each block for the sole purpose of applying the 0.2 and 0.3 g AuEq/t cutoffs to the appropriate materials within the optimized pits, as described above.

The total DeLamar project resources, which include the resources for both the DeLamar and Florida Mountain areas, are summarized in Table 14.19. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 14.19 Total DeLamar Project Gold and Silver Resources

Classification	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Measured	16,078,000	0.52	270,000	34.3	17,726,000
Indicated	156,287,000	0.42	2,106,000	19.7	98,788,000
Measured + Indicated	172,365,000	0.43	2,376,000	21.0	116,514,000
Inferred	28,266,000	0.38	343,000	13.5	12,240,000

1. Mineral Resources are comprised of all oxidized and transitional model blocks at a 0.2 g AuEq/t cutoff and all unoxidized blocks at a 0.3 g AuEq/t that lie within optimized pits
2. The effective date of the resource estimations is May 1, 2019
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability
4. Rounding may result in apparent discrepancies between tonnes, grade, and contained metal content

The gold and silver resources for the DeLamar and Florida Mountain areas are reported separately in Table 14.20 and Table 14.21, respectively.

Table 14.20 DeLamar Area Gold and Silver Resources

Classification	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Measured	14,481,000	0.51	238,000	36.4	16,942,000
Indicated	105,140,000	0.39	1,334,000	23.4	79,241,000
Measured + Indicated	119,621,000	0.41	1,572,000	25.1	96,183,000
Inferred	21,291,000	0.39	266,000	15.2	10,418,000

1. Mineral Resources are comprised of all oxidized and transitional model blocks at a 0.2 g AuEq/t cutoff and all unoxidized blocks at a 0.3 g AuEq/t that lie within optimized pits
2. The effective date of the DeLamar deposit DeLamar area resources is May 1, 2019
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability
4. Rounding may result in apparent discrepancies between tonnes, grade, and contained metal content

Table 14.21 Florida Mountain Area Gold and Silver Resources

Classification	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Measured	1,597,000	0.63	32,000	15.3	784,000
Indicated	51,147,000	0.47	772,000	11.9	19,547,000
Measured + Indicated	52,744,000	0.47	804,000	12.0	20,331,000
Inferred	6,975,000	0.34	77,000	8.1	1,822,000

1. Mineral Resources are comprised of all oxidized and transitional model blocks at a 0.2 g AuEq/t cutoff and all unoxidized blocks at a 0.3 g AuEq/t that lie within optimized pits
2. The effective date of the Florida Mountain deposit DeLamar area resources is May 1, 2019
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability
4. Rounding may result in apparent discrepancies between tonnes, grade, and contained metal content

The current mineral resources include only the modeled mineralization that was not mined during the historical underground and open-pit operations. The tonnage of the historical stopes and related workings modeled by MDA were also removed from the Florida Mountain resources.

The DeLamar project resources are classified according to the criteria presented in Table 14.22. The Measured and Indicated constraints for the DeLamar area are less restrictive than those for Florida Mountain due to the greater quantity of Integra drill data at DeLamar.

Table 14.22 Resource Classification Parameters

Area	Classification	Criteria
DeLamar	Measured	Minimum of 2 holes contributing composites, including at least 1 drilled by Integra, that lie within an average distance of 25 meters from the block
	Indicated	Minimum of 2 holes contributing composites that lie within an average distance of 40 meters from the block
	Inferred	all other blocks that qualify as resources
Florida Mountain	Measured	Minimum of 2 holes contributing composites, including at least 1 drilled by Integra, that lie within an average distance of 20 meters from the block
	Indicated	Minimum of 2 holes contributing composites that lie within an average distance of 20 meters from the block
	Inferred	all other blocks that meet the resource constraints

Although the authors are not expert with respect to any of the following aspects of the project, the authors are not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors not discussed in this report that could materially affect the potential development of the DeLamar project mineral resources as of the effective date of the report.

Figure 14.11 through Figure 14.16 are representative cross-sections showing the estimated block-model gold and silver grades, respectively, for the DeLamar area. These figures correspond to the mineral domain cross-sections presented in Figure 14.1 through Figure 14.6.

Figure 14.17 through Figure 14.20 are representative cross-sections showing the estimated block-model gold and silver grades, respectively, for the Florida Mountain area. These figures correspond to the Florida Mountain mineral domain cross-sections presented in Figure 14.7 through Figure 14.10.

Figure 14.11 Cross Section 1230 NW Showing Sullivan Gulch Block-Model Gold Grades

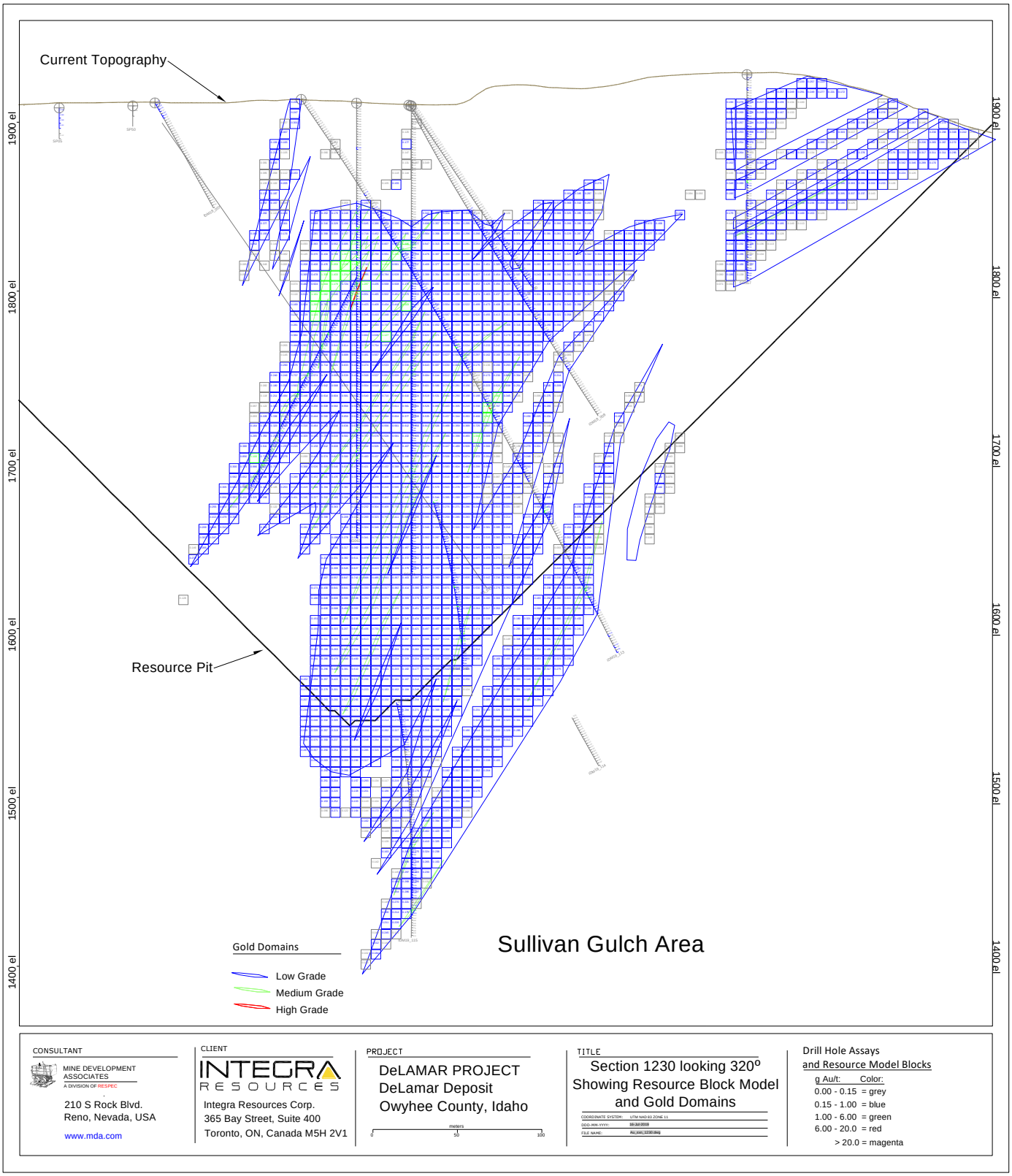


Figure 14.12 Cross Section 1230 NW Showing Sullivan Gulch Block-Model Silver Grades

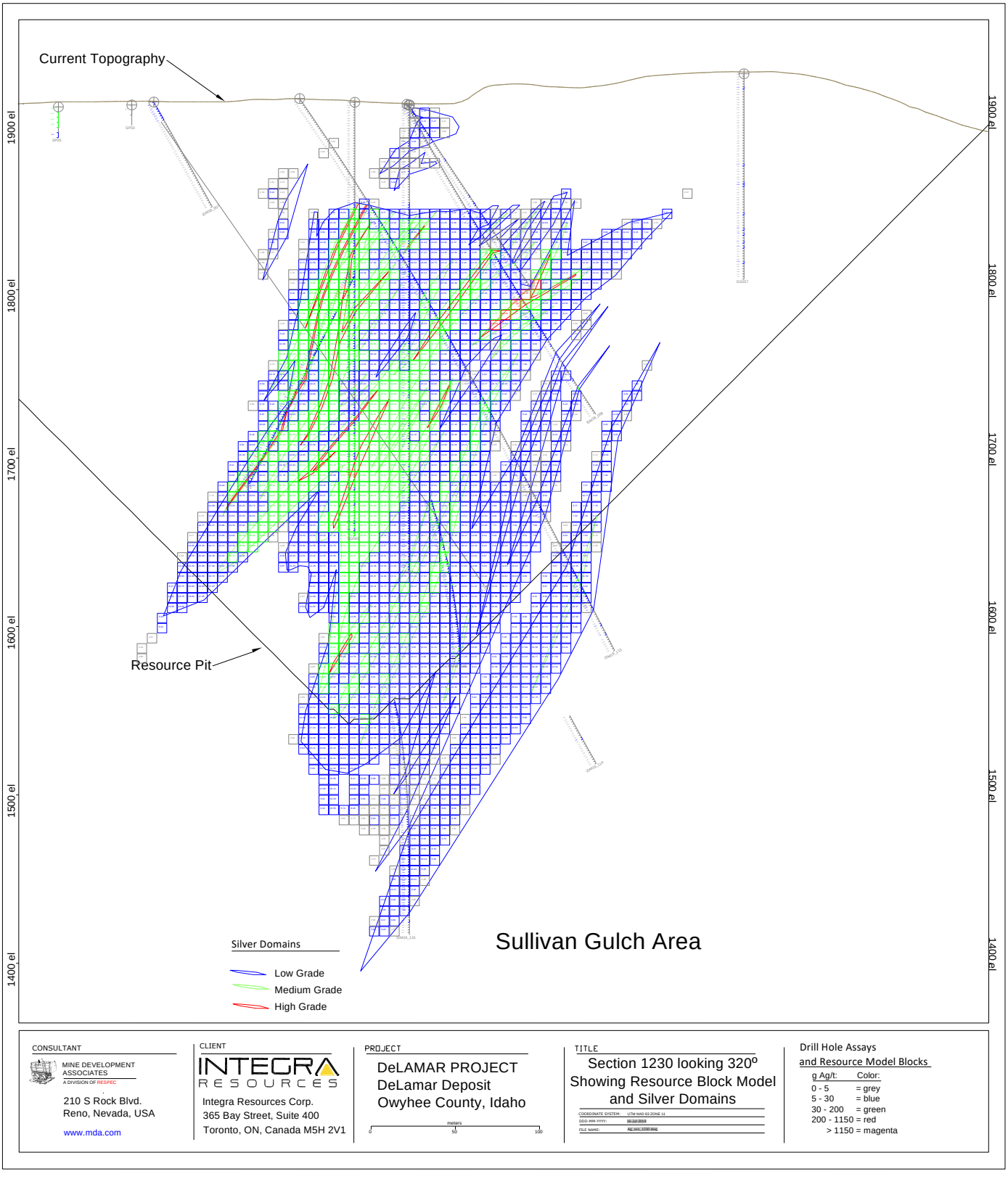


Figure 14.13 Cross Section 2010 NW Showing Sommercamp – Regan and N. DeLamar Block-Model Gold Grades

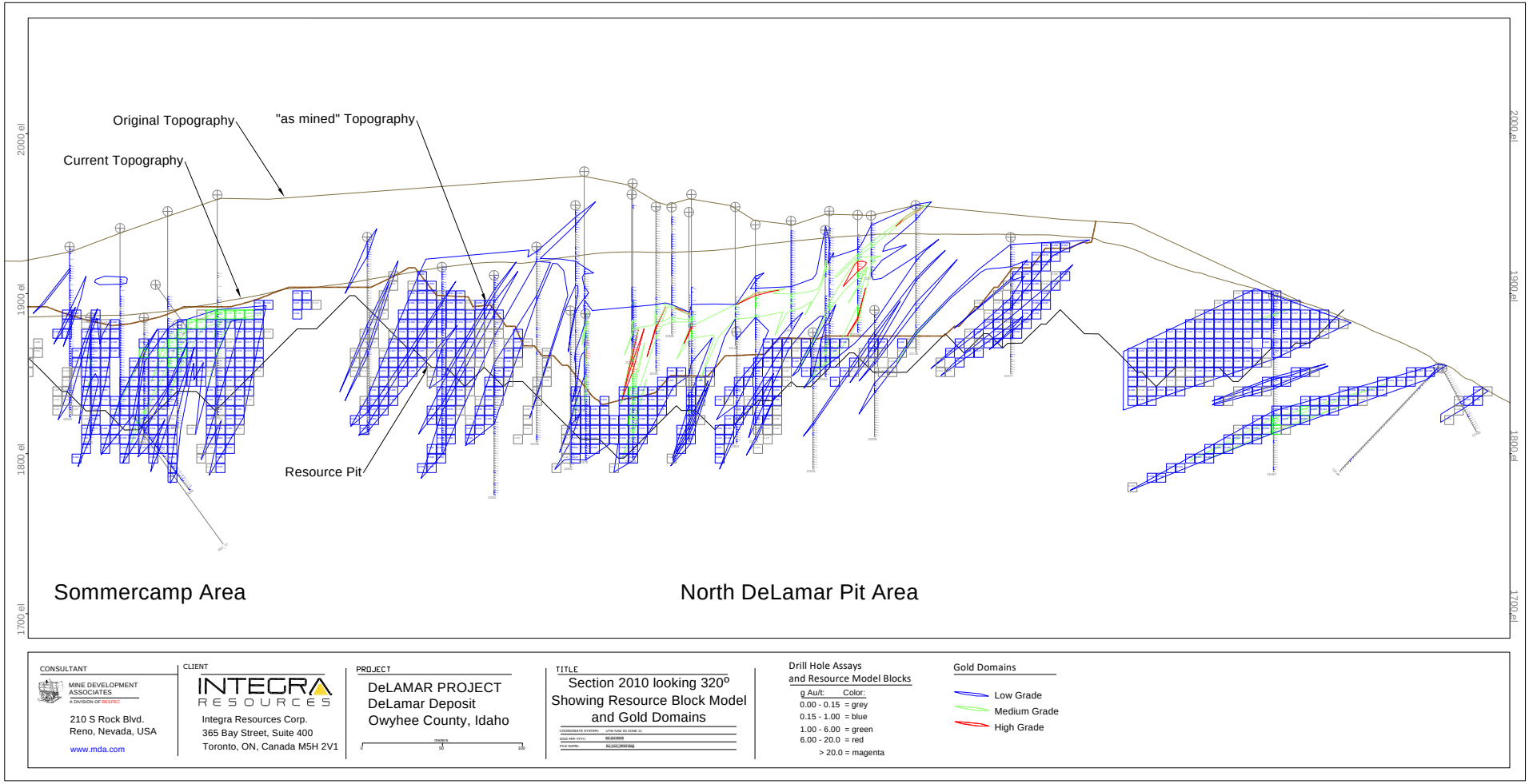


Figure 14.14 Cross Section 2010 NW Showing Sommercamp – Regan and N. DeLamar Block-Model Silver Grades

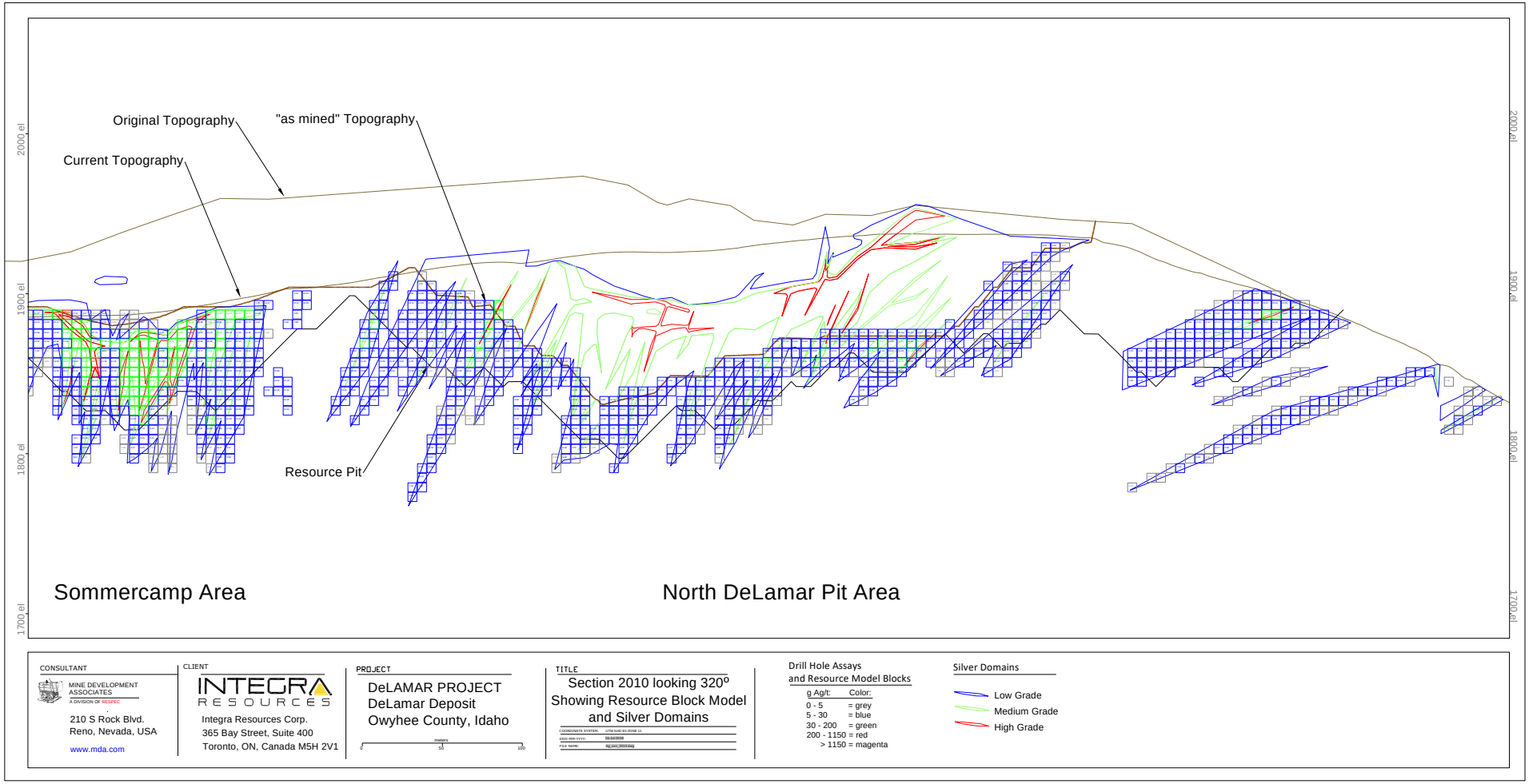


Figure 14.15 Cross Section 2790 NW Showing Glen Silver Block-Model Gold Grades

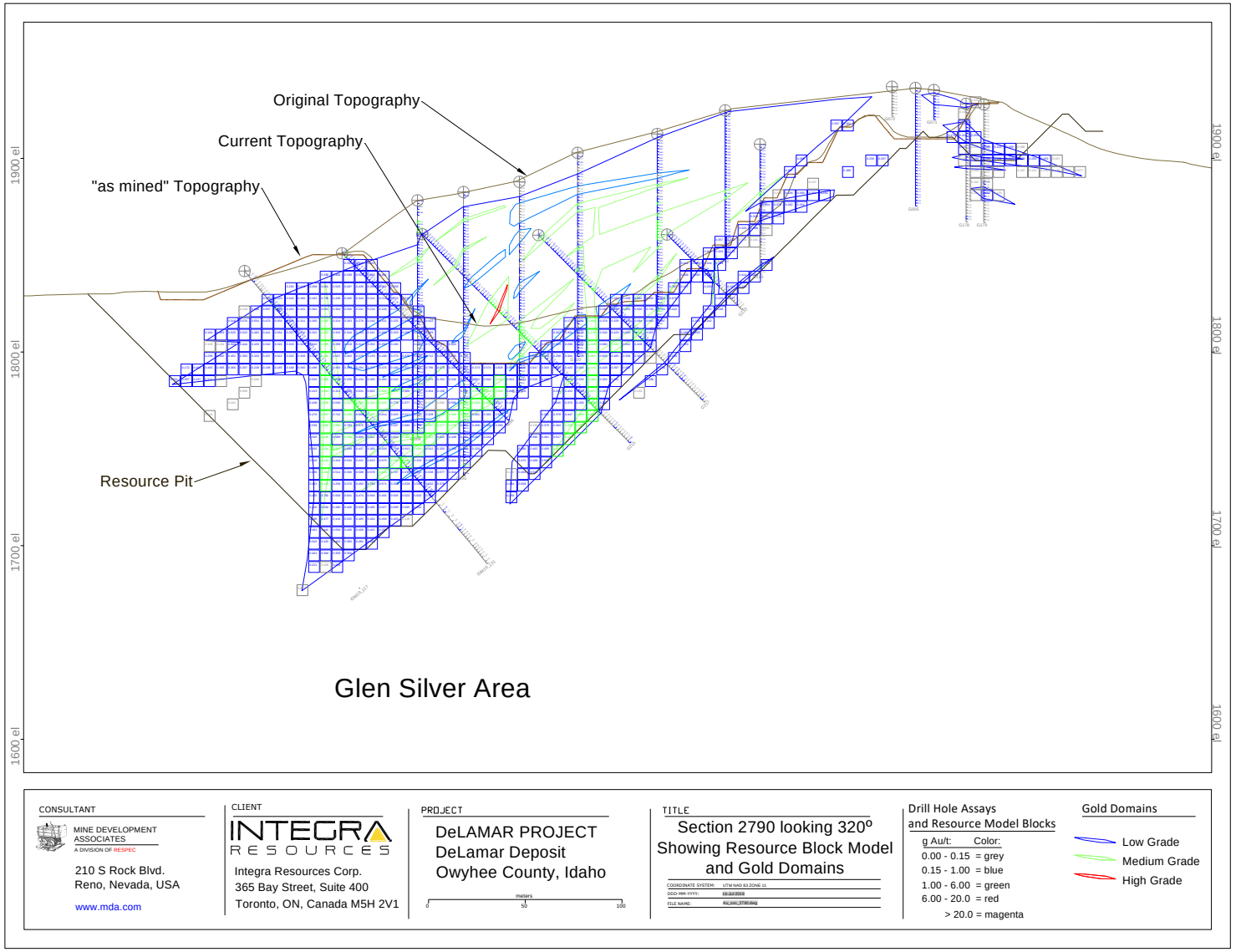


Figure 14.16 Cross Section 2790 NW Showing Glen Silver Block-Model Silver Grades

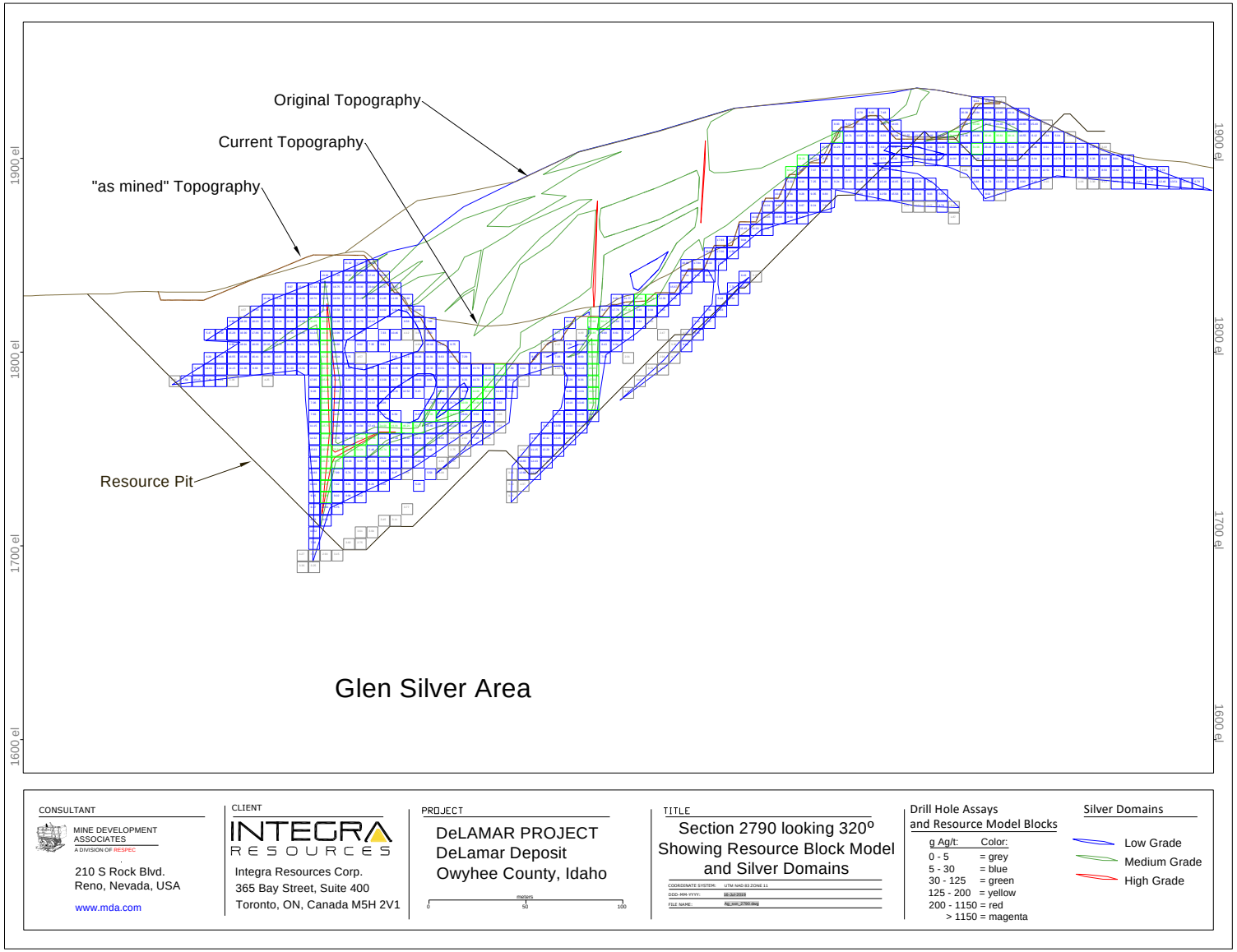


Figure 14.17 Cross Section 2830 N Showing Florida Mountain Block-Model Gold Grades

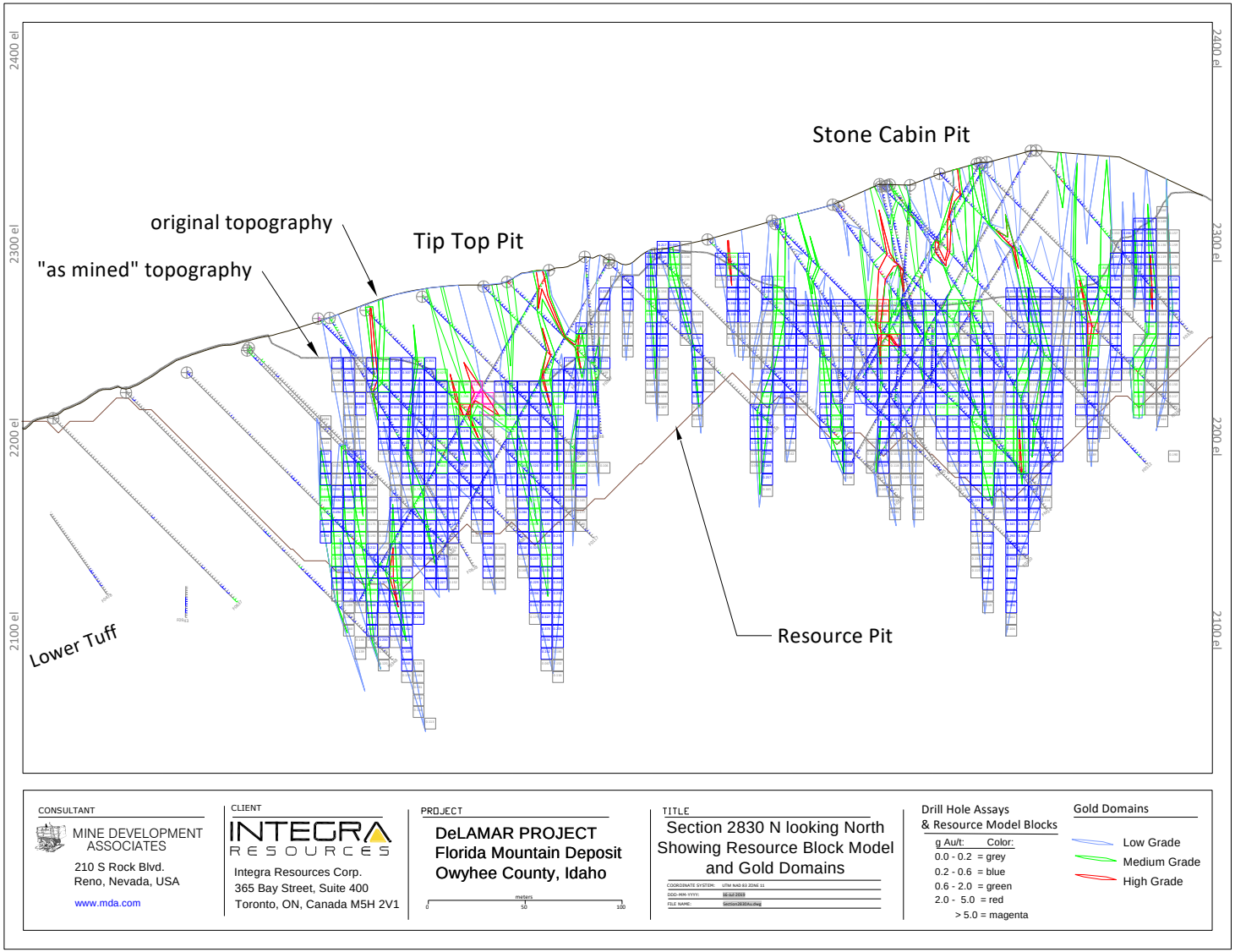


Figure 14.18 Cross Section 2830 N Showing Florida Mountain Block-Model Silver Grades

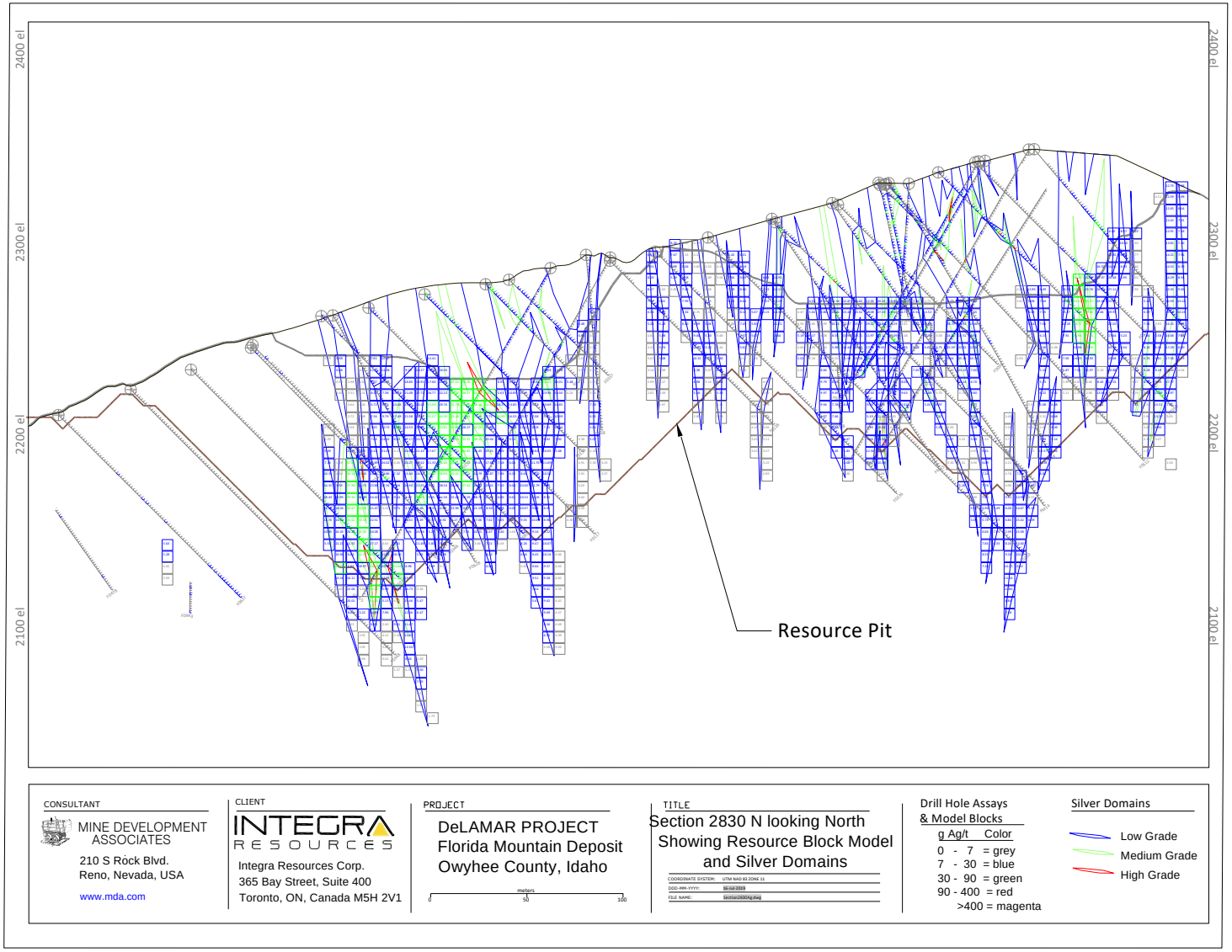


Figure 14.19 Cross Section 3280 N Showing Florida Mountain Block-Model Gold Grades

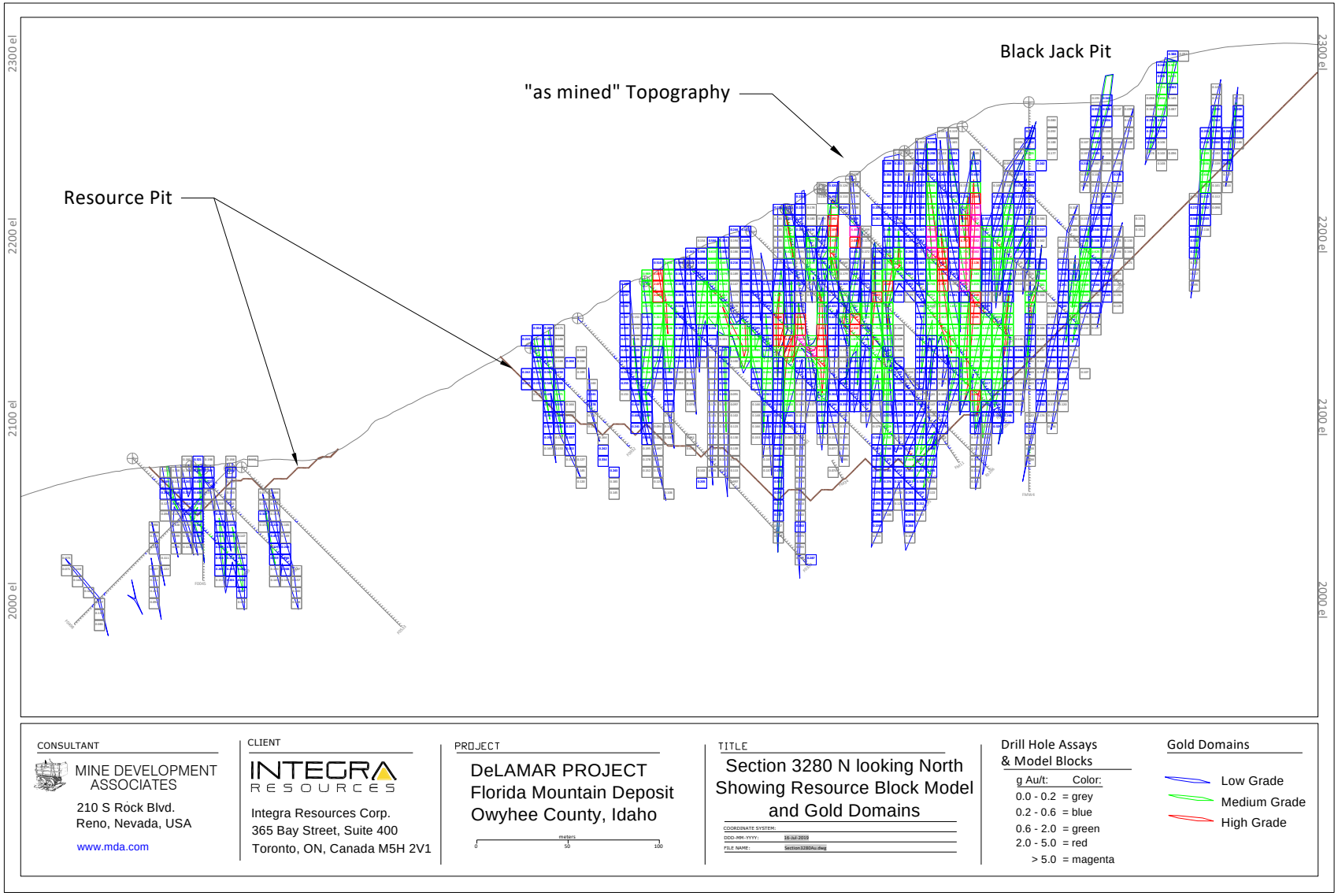
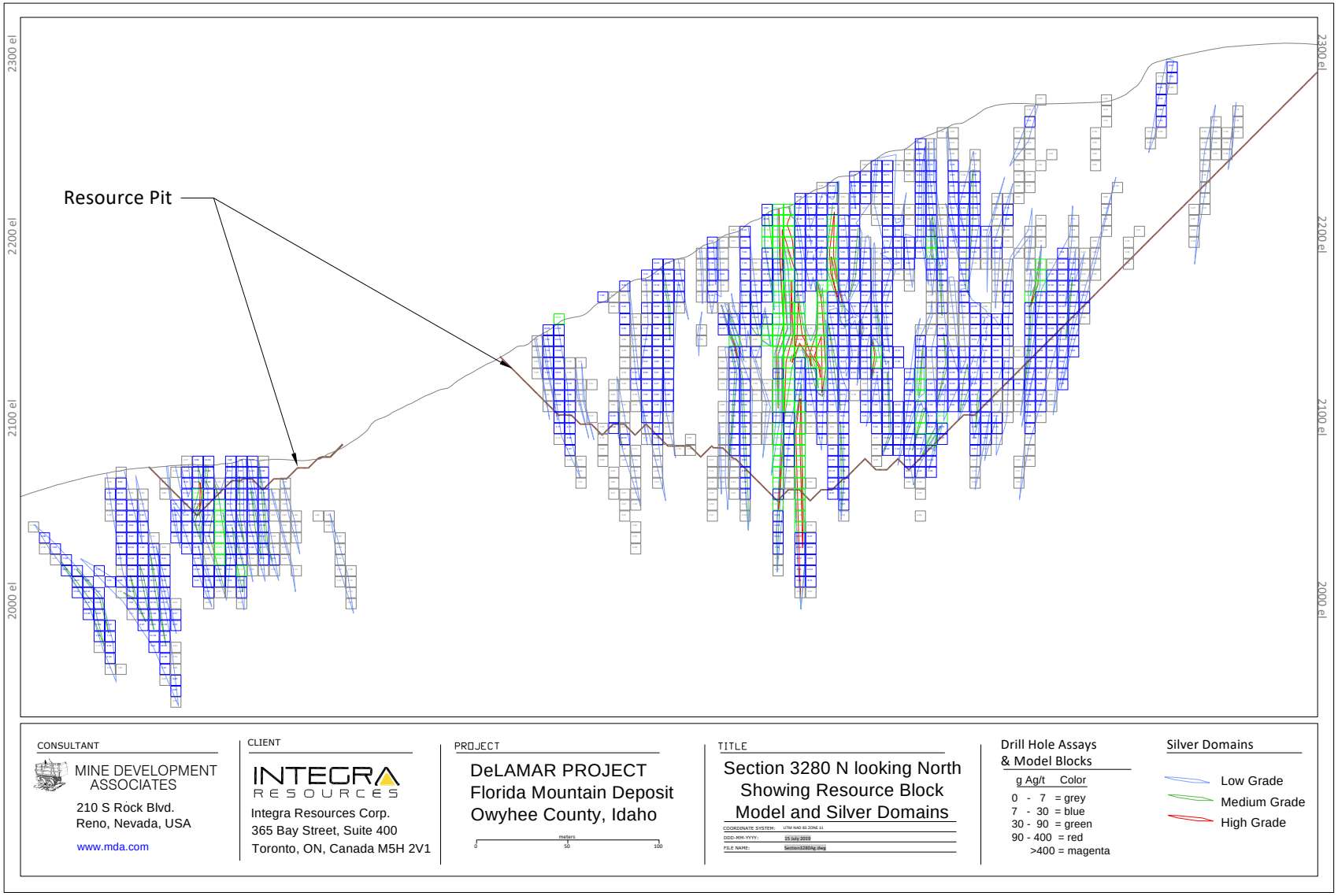


Figure 14.20 Cross Section 3280 N Showing Florida Mountain Block-Model Silver Grades



The modeled mineralization within the optimized pits that constrain the current DeLamar and Florida Mountain resources is tabulated at various cutoffs in Table 14.23 and Table 14.24, with the current resources highlighted in bold. This information is presented to provide grade-distribution data for each of the two resource areas, which allows for detailed assessments of the current project resources. The materials tabulated meet the requirement of reasonable prospects of economic extraction, as they are part of the current resources that are constrained as lying within optimized pits and at the resource cutoffs. As such, the mineralized materials tabulated at cutoffs higher than the resource cutoffs represent subsets of the current resources.

Table 14.23 Total Project In-Pit Oxidized and Transitional Mineralization at Various Cutoffs

	Measured + Indicated				
Cutoff g AuEq/t	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
0.20	***92,548,000***	***0.38***	***1,141,000***	***16.5***	***49,239,000***
0.30	65,085,000	0.46	972,000	20.0	41,820,000
0.40	43,360,000	0.56	782,000	24.2	33,771,000
0.50	28,389,000	0.67	616,000	29.0	26,477,000
0.75	11,030,000	1.02	361,000	40.5	14,377,000
1.00	5,609,000	1.37	247,000	49.2	8,866,000

	Inferred				
Cutoff g AuEq/t	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
0.20	***10,896,000***	***0.31***	***109,000***	***9.5***	***3,321,000***
0.30	6,242,000	0.39	79,000	11.8	2,368,000
0.40	3,518,000	0.47	54,000	14.0	1,580,000
0.50	1,838,000	0.57	33,000	17.0	1,004,000
0.75	381,000	0.90	11,000	22.7	278,000
1.00	116,000	1.31	5,000	27.8	104,000

Note: Rounding may cause apparent discrepancies.

Table 14.24 Total Project In-Pit Unoxidized Mineralization at Various Cutoffs

	Measured + Indicated				
Cutoff g AuEq/t	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
0.30	***79,817,000***	***0.48***	***1,235,000***	***26.2***	***67,275,000***
0.40	64,411,000	0.54	1,117,000	30.4	62,930,000
0.50	49,509,000	0.61	970,000	36.3	57,780,000
0.75	26,351,000	0.79	672,000	54.6	46,230,000
1.00	16,761,000	0.96	518,000	70.2	37,822,000
	Inferred				
Cutoff g AuEq/t	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
0.30	***17,369,000***	***0.42***	***235,000***	***16.0***	***8,920,000***
0.40	13,363,000	0.47	203,000	18.4	7,907,000
0.50	9,356,000	0.54	162,000	22.2	6,691,000
0.75	3,205,000	0.78	81,000	36.7	3,779,000
1.00	1,532,000	1.04	51,000	52	2,561,000

Note: Rounding may cause apparent discrepancies.

14.11 Discussion of Resource Modeling

The current DeLamar project resources include resources in the Indicated, and to a much lesser extent, Measured categories, which is a significant change from the prior resources, which were classified entirely as Inferred (Gustin and Weiss, 2017; 2018). This upgrade in classification is the consequence of: (*i*) the resource modeling is fully rectified three-dimensionally, as opposed to the two-dimensional cross-sectional modeling of the prior resource estimations; (*ii*) the geological support for the resource modeling has increased dramatically, due to Integra's detailed geologic interpretations that served as the base for the gold and silver domain modeling and the addition of detailed oxidation modeling; (*iii*) the historical data used in the resource modeling has undergone additional verification, most importantly by the consistency of Integra's drilling results with those of the historical drilling programs; (*iv*) the factored silver analyses are now fully understood and identified, and no mine-site silver analyses using the suspect analytical method are used in the resource estimations; (*v*) the as-mined pit topographies at the DeLamar area have been revised using high-confidence historical records; (*vi*) the historical specific-gravity measurements are better understood and are now complimented with measurements from Integra core samples; and (*vi*) the newly modeled oxidation states are coupled with Integra's new metallurgical testing results allow for the application of unique resource cutoffs linked to potential processing methods for the various oxidation states.

As discussed in Section 12.2.1, the mine-lab AA analyses systematically understated silver values, which the lab attributed to incomplete sample digestion. Mr. Gustin has verified this understatement, but in lieu of factoring these values as the historical mine operators did, the silver AA values were not used in the current resource grade interpolations. In order to evaluate the impact of this exclusion of silver data, which amount to 20% of all silver assays inside of the modeled silver domains at both DeLamar and Florida Mountain, a check estimation was run that included the AA silver analyses. At DeLamar, the inclusion of the understated silver values led to a loss of 4.7 million ounces of silver (4%

of the reported resource ounces) and 10,000 ounces of gold (0.5% of reported resource gold ounces). At Florida Mountain, the check estimate resulted in losses of 850,000 ounces of silver (4% or resource ounces) and 2,500 ounces of gold (0.4% of resource ounces). The pit optimizations used for the reported resources were applied to these check estimates; if new pit optimizations were to be run based on the models estimated with the AA silver data, slightly higher losses may result.

Consistent with assaying methods of the time period, some of the historical gold assays in the project databases were completed at a detection limit of 0.17 g Au/t (0.005 oz/ton). Current economic parameters for heap leach processing can approach this detection-limit grade. Possible effects on the resource block model imparted by the lowered precision of some of the gold assay data at grades at and near the detection limit, if any, are difficult to predict due to the averaging that occurs during grade interpolation. In any event, any future potential mining operation that incorporates very low cutoff grades should be aware of the presence of these assays in the historical database.

Some of the samples assayed by methods with a 0.17 g Au/t detection limit also have lowered precision at higher grades. These samples have gold results variably reported at a precisions of 0.17 or 0.34 g Au/t (0.005 or 0.010 oz/ton). A total of 14% of the un-composited assays coded to the gold mineral domains in the DeLamar area, and 1.5% of those at Florida Mountain, are characterized by these lower-precision analyses.

The drilling that forms the basis of the resource estimations was done primarily by RC and conventional-rotary methods, which can be affected by down-hole contamination. As discussed elsewhere in this report, a small quantity of drill intervals in which down-hole contamination was suspected were excluded from use in the estimations. However, potentially contaminated samples may remain in the data used in the estimations, although Mr. Gustin believes the possible inclusion of such samples is not a material issue at DeLamar or Florida Mountain.

The historical underground stopes in the DeLamar area have effectively been mined out by the historical open-pit mining operations, and although some of the related developmental crosscuts, etc., remain within the resources, their volumes are insignificant. At Florida Mountain, stopes and related workings along the Black Jack – Trade Dollar vein system, which were modeled by MDA, extend upwards into the Florida Mountain resource model. A total of 204,000 tonnes of material lying within the modeled underground solids that would have otherwise been part of the Florida Mountain reported resources were removed from the resources.

Within the limits of the current Florida Mountain deposit resources, it is not uncommon for drill holes to have markedly different grades than adjacent holes. Mr. Gustin believes this variability is properly represented in the resource model due to both the explicit modeling of the gold and silver domains and the tight drill spacing at Florida Mountain, where a high percentage of resource blocks lie within an average distance of 20 meters from a minimum of two drill holes.

15.0 MINERAL RESERVE ESTIMATES

There are no estimated mineral reserves as of the date of this report.

16.0 MINING METHODS

The PEA presented in this report considers open-pit mining of the DeLamar and Florida Mountain gold-silver deposits. Note that a PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves. There is no certainty that the economic results of the PEA will be realized.

The methodology used for mine planning to define the economics for the PEA includes:

- Define assumptions for the economic parameters;
- Define geometric parameters and constraints;
- Run pit optimizations;
- Define road and ramp parameters;
- Create pit designs;
- Create dump designs;
- Produce mine and process production schedules;
- Define personnel and equipment requirements;
- Estimate mining costs; and
- Perform an economic analysis.

Section 16.0 summarizes the above topics, except for the mining cost estimates which are discussed in Section 21, and the economic analysis discussed in Section 22.

16.1 Economic Parameters

Economic parameters were used to generate optimized pits using a Lerchs-Grossman algorithm within Whittle™ software (Version 4.7). The economic parameters include mining costs, processing costs, general and administrative costs ("G&A"), refining costs, royalties, and metal recoveries. Mine planning is an iterative process, and initial costs and recoveries were assumed to determine how large pits would be. The economic parameters were refined as concepts were developed on how material would be processed from the two separate deposits. The methods for processing that were determined include:

- Use of crushing and cyanide heap leaching for oxide and transition material from DeLamar and Florida Mountain; and
- Using flotation milling for higher sulfide material from Florida Mountain;

The economic parameters used are shown in Table 16.1. The overall initial process rate is assumed to be 15,000 tonnes per day or 5,250,000 tonnes per year. This initial process rate is used here to convert the fixed G&A component to a cost per tonne for the purpose of pit optimization. Note that the final process rates used in the PEA were 27,000 tonnes per day, or 9,720,000 tonnes per year, for heap

leaching and 2,000 tonnes per day, or 720,000 tonnes per year, for flotation milling of the Florida Mountain unoxidized material. Leaching of Florida Mountain material would include secondary crushing, and leaching of the DeLamar material would involve tertiary crushing and agglomeration. The G&A cost is later applied as a fixed cost in the cash-flow model.

Table 16.1 DeLamar and Florida Mountain Economic Parameters

	DeLamar	Florida Mnt	Units
Mining	$ 2.20	$ 2.20	$/t Mined
Leach Process	$ 3.35	$ 3.00	$/t Processed
Float Process	NA	$ 16.00	$/t Processed
G&A Cost	$ 4,000	$ 4,000	K $/year
Tonnes per Day	15,000	15,000	TPD
Tonnes per Year	5,250	5,250	K TPY (350)
G&A Cost	$ 0.76	$ 0.76	$/t Processed
Refine Lch - Au	$ 5.00	$ 5.00	$/oz Au Prod
Refine Lch - Ag	$ 0.50	$ 0.50	$/oz Ag Prod
NSR Royalty	1%	0%	

Royalties were applied by royalty area or region as provided by Integra. These are described in Section 4.3.

Recoveries were applied based on recommendations by Mr. Jack McPartland, as summarized in Section 13.0. Recoveries are shown in Table 16.2. The oxide and transition recoveries assume crushed leaching for oxide and transition material, and flotation milling for Florida Mountain unoxidized material.

Table 16.2 DeLamar and Florida Mountain Recoveries

	DeLamar		Florida Mountain		
	Oxide	Trans	Oxide	Trans	UnOx
Rec - Au	80%	75%	90%	85%	90%
Rec - Ag	30%	30%	40%	40%	80%

16.2 Cutoff Grades

Cutoff grades were calculated based on the economic parameters shown in Table 16.1. These were calculated for the different deposits and material types for the various potential processing methods. Crush and agglomerate, and toll processing cutoff grades were all calculated as internal break-even cutoffs. The internal cutoff grade calculation eliminates the mining cost in the calculation. The pit designs are based on economical pits and the materials inside of the pits are assumed to be mined whether the material is waste or ore. Thus, the decision on whether to process the material is made at the point where the truck needs to turn either to the waste dump or the process facility. The mining cost is therefore a sunk cost. The basic equation for the cutoff grade calculation is shown in Equation 1.

Equation 1 Breakeven Cutoff Grade Calculation (g Au/t)

$$\frac{Costs}{\left(\frac{Au\$}{oz} - RefCst\right) * \frac{1oz}{31.10348g} * (1 - Roy\%) * Rec\%}$$

Where costs are all processing costs plus G&A costs in $/t, RefCst is the refining cost in $/oz gold produced, Roy% is the NSR royalty, and Rec% is the recommended recovery.

Cutoff grades are calculated in terms of g Au/t, and are applied to a gold equivalent grade. The calculation for the gold equivalent grade for the PEA is shown in Equation 2. Table 16.3 shows the resulting gold equivalent factors used.

Equation 2 Gold Equivalent Calculation

$$Au_{Equivalent} = Au_{\frac{g}{t}} + {}^{Ag_{\frac{g}{t}}}\Big/_{AuEq_{Fact}}$$

Where:

$$AuEq_{Fact} = \frac{Au_{Price}}{Ag_{Price}} * \frac{Au_{Recovery}}{Ag_{Recovery}}$$

Table 16.3 DeLamar and Florida Mountain AuEq Factors

	Oxide	Trans	UnOx
DeLamar	206	193	NA
Florida Mountain	174	164	87

Table 16.4 shows the cutoff grades for DeLamar and Florida Mountain at various gold prices. The economic analysis to determine the ultimate pit has been done based on a gold price of $1,275 per ounce Au. Because these cutoff grades are fairly low with respect to assay detection levels, a minimum cutoff grade for leaching of 0.20 g AuEq/t has been applied. A cutoff grade of 0.46 g AuEq/t was applied for the unoxidized material from Florida Mountain.

Table 16.4 DeLamar and Florida Mountain Cutoff Grades (g Au/t)

	DeLamar		Florida Mountain		
$/oz Au	Oxide	Trans	Oxide	Trans	UnOx
$ 1,200	0.14	0.14	0.11	0.12	0.48
$ 1,225	0.13	0.14	0.11	0.11	0.47
$ 1,250	0.13	0.14	0.10	0.11	0.47
$ 1,275	0.13	0.14	0.10	0.11	0.46
$ 1,300	0.12	0.13	0.10	0.11	0.45
$ 1,325	0.12	0.13	0.10	0.10	0.44
$ 1,350	0.12	0.13	0.10	0.10	0.43
$ 1,375	0.12	0.13	0.09	0.10	0.42
$ 1,400	0.12	0.12	0.09	0.10	0.42

16.3 Geometric Parameters

Geometric parameters include slope parameters and land constraints. The author is not aware of any geotechnical studies that have been completed to date for either deposit. Thus, a simple 45° overall slope is assumed for pit optimization and design.

The other geometric parameters include royalty boundaries. These were flagged into the block model and exported into Whittle. The appropriate NSR royalty values were assigned in Whittle to account for the reduction of revenues due to required royalty payments.

16.4 Pit Optimization

Pit optimizations were run using Whittle™ software (version 4.7). Inputs into Whittle included the resource block model along with the economic and geometric parameters previously discussed. Each deposit was run separately, and ultimate pit shells were selected from the Whittle results for final designs. For DeLamar and Florida Mountain, additional pit shells were considered for guidance of interior pit phases.

The selections of ultimate pits and pit phases were done as a two-step process. The first step was to optimize a set of pit shells based on varying a revenue factor. This was done in Whittle using a Lerchs-Grossman ("LG") algorithm. The revenue factor was multiplied by the recovered ounces and the metal prices, essentially creating a nested set of pit shells based on different metal prices. Revenue factors for each of the deposits were varied from 0.30 to 2.0 in increments of 0.025 with a base price of $1,000 per ounce of gold, so the resulting pit shells represent gold prices from $300 to $2,000 per ounce in increments of $25.00. This has the potential of generating up to 69 different pit shells that can be used for analysis.

The second step of the process was to use the Pit by Pit ("PbP") analysis tool in Whittle to generate a discounted operating cash flow (note that capital is not included). This uses a rough scheduling by pit phase for each pit shell to generate the discounted value for the pit. The program develops three

different discounted values: best, worst, and specified. The best-case value uses each of the pit shells as pit phases or pushbacks. For example, when evaluating pit 20, there would be 19 pushbacks mined prior to pit 20, and the resulting schedule takes advantage of mining more valuable material up front to improve the discounted value. Evaluating pit 21 would have 20 pushbacks; pit 22 would have 21 pushbacks and so on. Note that this is not a realistic case as the incremental pushbacks would not have enough mining width between them to be able to mine appropriately, but this does help to define the maximum potential discounted operating cash flow.

The worst case does not use any pushbacks in determining the discounted value for each of the pit shells. Thus, each pit shell is evaluated as if mining a single pit from top to bottom. This does not provide the advantage of mining more valuable material first, so it generally provides a lower discounted value than that of the best case.

The specified case allows the user to specify pit shells to be used as pushbacks and then schedules the pushbacks and calculates the discounted cash flow. This is more realistic than the base case as it allows for more mining width, though the final pit design will have to ensure that appropriate mining width is available. The specified case has been used for each of the two DeLamar project mines to determine the ultimate pit limits to design to, as well as to specify guidelines for designing pit phases.

16.4.1 DeLamar Pit Optimization

The previously discussed parameters were used along with gold prices varying from $300 to $2,000 per ounce to create the pit optimization results for the DeLamar deposit. These results are shown in Table 16.5 in $100 gold price increments with the addition of a $1,275 and $1,350 pit shell which are highlighted in the table. The $1,250 price was used to determine the ultimate pit limits for design. The $1,350 price was used for the PEA economic evaluation as discussed in Section 22.

Table 16.6 shows the PbP results for pit shells 29 through 45. The highlighted pit is the one that maximizes the discounted operating cash flow and is used as the basis for the pit designs. Figure 16.1 shows the results of the DeLamar PbP analysis graphically.

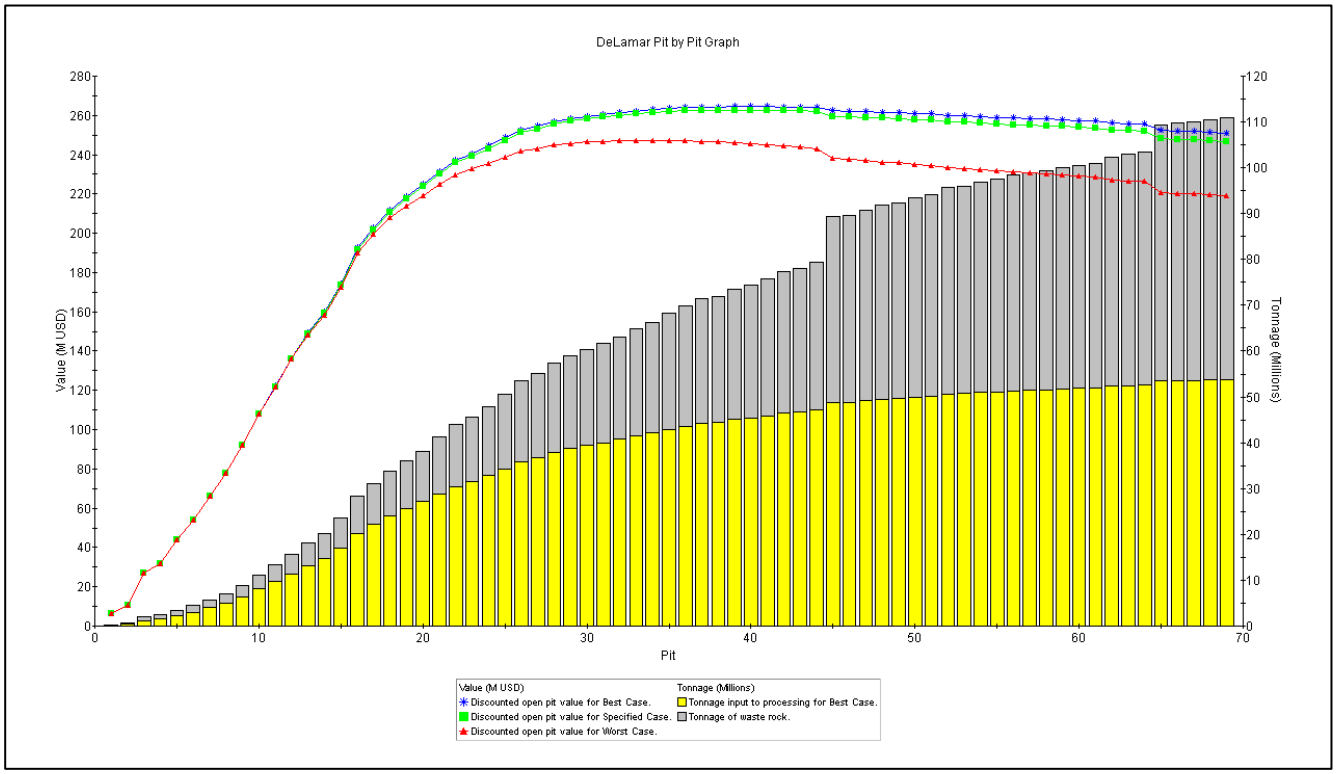
Table 16.5 DeLamar Pit Optimization Results

			Total Processed					Waste	Total	Strip
Pit	$/oz Au	$/oz Ag	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	K Tonnes	K Tonnes	Ratio
1	$ 300	$ 3.88	155	1.15	6	50.65	252	169	324	1.09
5	$ 400	$ 5.18	1,894	0.73	44	41.91	2,552	1,604	3,498	0.85
9	$ 500	$ 6.47	5,659	0.57	104	32.53	5,918	3,254	8,913	0.58
13	$ 600	$ 7.76	12,054	0.48	188	27.78	10,765	6,006	18,059	0.50
17	$ 700	$ 9.06	20,925	0.43	287	23.99	16,138	10,086	31,012	0.48
21	$ 800	$ 10.35	28,202	0.39	356	22.12	20,059	12,989	41,191	0.46
25	$ 900	$ 11.65	34,320	0.37	409	20.93	23,091	16,299	50,619	0.47
29	$ 1,000	$ 12.94	38,773	0.36	448	20.23	25,223	20,214	58,987	0.52
33	$ 1,100	$ 14.24	41,463	0.35	470	19.89	26,509	23,320	64,782	0.56
37	$ 1,200	$ 15.53	44,188	0.35	492	19.48	27,670	27,162	71,350	0.61
40	$ 1,275	$ 16.50	45,458	0.34	501	19.29	28,189	28,938	74,397	0.64
41	$ 1,300	$ 16.82	45,930	0.34	505	19.23	28,398	29,865	75,795	0.65
43	$ 1,350	$ 17.47	46,734	0.34	511	19.14	28,756	31,341	78,075	0.67
45	$ 1,400	$ 18.12	48,701	0.34	526	20.04	31,385	40,591	89,292	0.83
49	$ 1,500	$ 19.41	49,628	0.33	533	19.96	31,848	42,745	92,373	0.86
53	$ 1,600	$ 20.71	50,736	0.33	541	19.81	32,306	45,328	96,063	0.89
57	$ 1,700	$ 22.00	51,474	0.33	546	19.76	32,699	47,507	98,980	0.92
61	$ 1,800	$ 23.29	52,017	0.33	550	19.66	32,883	48,987	101,004	0.94
65	$ 1,900	$ 24.59	53,496	0.33	561	19.58	33,673	55,903	109,400	1.04
69	$ 2,000	$ 25.88	53,846	0.33	564	19.52	33,800	57,139	110,985	1.06

Table 16.6 DeLamar Pit by Pit Results

	Material Processed					Waste	Total	Strip	Op Cash Flow (M USD)			Years Process
Pit	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	K Tonnes	K Tonnes	Ratio	Best	Spec	Worst	Leach
29	38,773	0.36	448	20.23	25,223	20,214	58,987	0.52	$ 258.47	$ 257.17	$ 245.79	5.39
30	39,414	0.36	453	20.17	25,563	20,961	60,375	0.53	$ 259.59	$ 258.27	$ 246.39	5.47
31	40,037	0.36	458	20.08	25,848	21,571	61,608	0.54	$ 260.47	$ 259.13	$ 246.76	5.56
32	40,759	0.35	464	19.95	26,149	22,352	63,111	0.55	$ 261.41	$ 260.03	$ 247.09	5.66
33	41,463	0.35	470	19.89	26,509	23,320	64,782	0.56	$ 262.23	$ 260.82	$ 247.27	5.76
34	42,171	0.35	475	19.77	26,804	24,117	66,288	0.57	$ 262.86	$ 261.40	$ 247.27	5.86
35	42,958	0.35	482	19.65	27,135	25,375	68,333	0.59	$ 263.49	$ 261.94	$ 247.09	5.97
36	43,631	0.35	487	19.56	27,445	26,348	69,979	0.60	$ 263.94	$ 262.36	$ 247.00	6.06
37	44,188	0.35	492	19.48	27,670	27,162	71,350	0.61	$ 264.23	$ 262.62	$ 246.86	6.14
38	44,468	0.35	494	19.44	27,787	27,549	72,017	0.62	$ 264.33	$ 262.70	$ 246.72	6.18
39	45,065	0.34	498	19.34	28,021	28,372	73,437	0.63	$ 264.45	$ 262.76	$ 246.19	6.26
40	45,458	0.34	501	19.29	28,189	28,938	74,397	0.64	$ 264.48	$ 262.75	$ 245.82	6.31
41	45,930	0.34	505	19.23	28,398	29,865	75,795	0.65	$ 264.45	$ 262.66	$ 245.25	6.38
42	46,429	0.34	509	19.19	28,646	30,855	77,285	0.66	$ 264.34	$ 262.48	$ 244.53	6.45
43	46,734	0.34	511	19.14	28,756	31,341	78,075	0.67	$ 264.24	$ 262.34	$ 244.06	6.49
44	47,171	0.34	514	19.15	29,049	32,274	79,445	0.68	$ 264.02	$ 262.03	$ 243.18	6.55
45	48,701	0.34	526	20.04	31,385	40,591	89,292	0.83	$ 262.31	$ 259.58	$ 238.08	6.76

Figure 16.1 DeLamar Pit by Pit Graph



16.4.2 Florida Mountain Pit Optimization

Florida Mountain pit optimizations also used the parameters previously discussed and the optimizations were completed with gold prices varying from $300 to $2,000 per ounce to create 69 pit shells. These results are shown in Table 16.7 in $100 gold price increments with the addition of a $1,275 and $1,350 pit shell, which are highlighted in the table.

Table 16.8 shows the PbP results for pit shells 29 through 45. The highlighted pit is the one that maximizes the discounted operating cash flow and is used as the basis for the pit designs. Figure 16.2 shows the results of the Florida Mountain PbP analysis graphically.

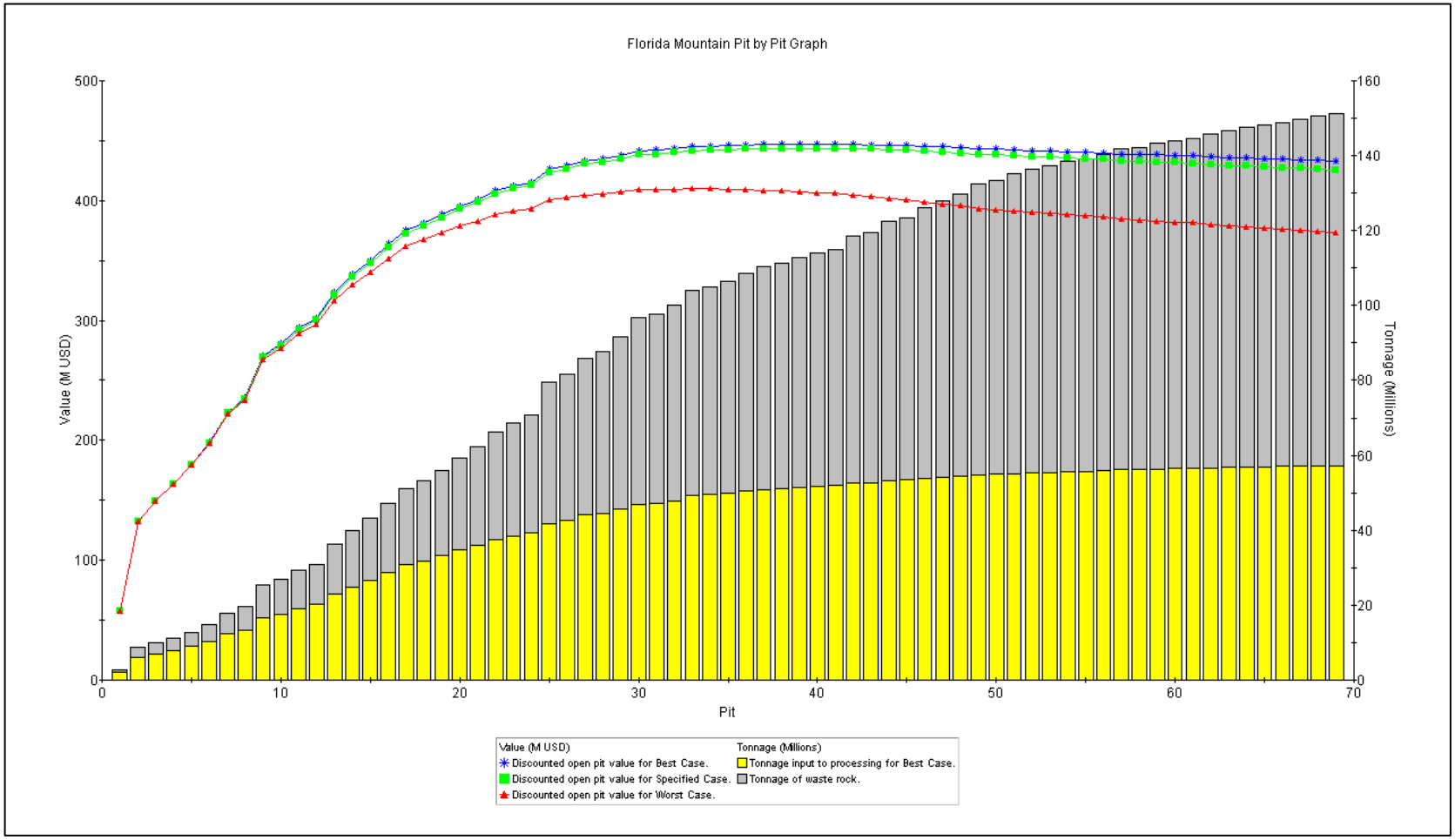
Table 16.7 Florida Mountain Pit Optimization Results

			Total Processed					Waste	Total	Strip
Pit	$/oz Au	$/oz Ag	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	K Tonnes	K Tonnes	Ratio
1	$ 300	$ 3.88	1,296	1.20	50	22.25	927	1,332	2,628	1.03
5	$ 400	$ 5.18	6,316	0.88	178	20.51	4,166	6,493	12,809	1.03
9	$ 500	$ 6.47	13,267	0.71	302	16.39	6,992	12,082	25,349	0.91
13	$ 600	$ 7.76	20,846	0.60	400	13.91	9,319	15,332	36,178	0.74
17	$ 700	$ 9.06	30,327	0.52	509	12.69	12,370	20,712	51,039	0.68
21	$ 800	$ 10.35	35,493	0.50	571	12.36	14,109	26,680	62,172	0.75
25	$ 900	$ 11.65	41,055	0.49	648	12.10	15,971	38,443	79,498	0.94
29	$ 1,000	$ 12.94	44,989	0.48	698	11.94	17,266	46,537	91,526	1.03
33	$ 1,100	$ 14.24	48,575	0.48	745	11.90	18,587	55,372	103,947	1.14
37	$ 1,200	$ 15.53	50,546	0.47	770	11.86	19,273	59,718	110,265	1.18
40	$ 1,275	$ 16.50	51,794	0.47	785	11.85	19,728	62,265	114,059	1.20
41	$ 1,300	$ 16.82	52,102	0.47	789	11.84	19,838	62,898	115,000	1.21
43	$ 1,350	$ 17.47	53,142	0.47	805	11.83	20,211	66,410	119,553	1.25
45	$ 1,400	$ 18.12	54,155	0.47	819	11.84	20,618	69,273	123,428	1.28
49	$ 1,500	$ 19.41	56,263	0.47	847	11.77	21,296	76,221	132,484	1.35
53	$ 1,600	$ 20.71	57,753	0.47	865	11.75	21,824	79,435	137,188	1.38
57	$ 1,700	$ 22.00	59,396	0.46	882	11.69	22,317	82,536	141,932	1.39
61	$ 1,800	$ 23.29	60,623	0.46	894	11.62	22,649	83,840	144,462	1.38
65	$ 1,900	$ 24.59	62,051	0.46	908	11.55	23,045	86,210	148,261	1.39
69	$ 2,000	$ 25.88	63,305	0.45	919	11.47	23,347	87,876	151,181	1.39

Table 16.8 Florida Mountain Pit by Pit Results

	Floatation					Leaching					Total Processed					Waste	Total	Strip	Op Cash Flow (M USD)			Years of Processing	
Pit	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	K Tonnes	K Tonnes	Ratio	Best	Spec	Worst	Floatation	Leach
29	2,885	0.96	89	18.17	1,685	42,807	0.45	618	11.47	15,787	45,692	0.48	708	11.89	17,472	45,834	91,526	1.00	$ 437.62	$ 435.17	$ 407.20	4.01	5.95
30	3,322	0.94	100	18.62	1,989	43,577	0.45	627	11.41	15,986	46,899	0.48	727	11.92	17,975	50,003	96,902	1.07	$ 441.33	$ 438.37	$ 408.83	4.61	6.05
31	3,387	0.93	102	18.74	2,041	43,821	0.45	629	11.40	16,058	47,208	0.48	731	11.92	18,099	50,595	97,803	1.07	$ 441.92	$ 438.92	$ 409.17	4.70	6.09
32	3,542	0.92	105	18.63	2,122	44,387	0.44	635	11.35	16,194	47,929	0.48	740	11.89	18,316	52,190	100,119	1.09	$ 443.17	$ 440.04	$ 409.57	4.92	6.16
33	3,744	0.90	109	18.43	2,218	45,467	0.44	644	11.33	16,559	49,211	0.48	753	11.87	18,777	54,736	103,947	1.11	$ 444.98	$ 441.85	$ 410.29	5.20	6.31
34	3,819	0.90	110	18.47	2,268	45,710	0.44	647	11.32	16,630	49,529	0.48	757	11.87	18,899	55,533	105,062	1.12	$ 445.39	$ 442.27	$ 409.93	5.30	6.35
35	3,941	0.89	113	18.37	2,328	45,935	0.44	649	11.30	16,695	49,876	0.47	761	11.86	19,024	56,503	106,379	1.13	$ 445.83	$ 442.69	$ 409.69	5.47	6.38
36	4,066	0.88	116	18.23	2,383	46,330	0.44	653	11.28	16,800	50,396	0.47	768	11.84	19,183	58,075	108,472	1.15	$ 446.33	$ 443.16	$ 409.19	5.65	6.43
37	4,165	0.88	118	18.20	2,437	46,674	0.44	656	11.28	16,922	50,839	0.47	774	11.84	19,359	59,425	110,265	1.17	$ 446.63	$ 443.45	$ 408.58	5.78	6.48
38	4,275	0.87	120	18.22	2,504	46,815	0.44	657	11.27	16,966	51,090	0.47	777	11.85	19,470	60,207	111,297	1.18	$ 446.77	$ 443.56	$ 408.17	5.94	6.50
39	4,397	0.86	122	18.21	2,575	47,085	0.44	660	11.26	17,047	51,482	0.47	782	12	19,622	61,315	112,797	1.19	$ 446.88	$ 443.65	$ 407.37	6.11	6.54
40	4,487	0.86	123	18.13	2,616	47,307	0.44	662	11.25	17,112	51,794	0.47	785	11.85	19,728	62,265	114,059	1.20	$ 446.92	$ 443.64	$ 406.49	6.23	6.57
41	4,569	0.85	125	18.18	2,671	47,422	0.43	663	11.24	17,137	51,991	0.47	788	11.85	19,807	63,009	115,000	1.21	$ 446.89	$ 443.58	$ 406.07	6.35	6.59
42	4,860	0.85	132	18.07	2,823	47,655	0.43	665	11.22	17,196	52,515	0.47	797	11.86	20,019	65,991	118,506	1.26	$ 446.63	$ 443.13	$ 404.63	6.75	6.62
43	4,943	0.84	134	18.04	2,867	47,822	0.43	667	11.21	17,242	52,764	0.47	800	11.85	20,109	66,788	119,553	1.27	$ 446.50	$ 442.95	$ 403.99	6.86	6.64
44	5,255	0.83	140	18.06	3,051	47,975	0.43	668	11.20	17,281	53,231	0.47	808	11.88	20,332	69,187	122,417	1.30	$ 446.03	$ 442.27	$ 401.46	7.30	6.66
45	5,359	0.82	142	18.08	3,116	48,084	0.43	669	11.20	17,308	53,442	0.47	811	11.89	20,423	69,986	123,428	1.31	$ 445.81	$ 441.96	$ 400.64	7.44	6.68

Figure 16.2 Florida Mountain Pit by Pit Graph



16.5 Road and Ramp Design

Road designs have been completed for the PEA to allow primary access for people, equipment, and consumables to the site. This includes haul roads between the designed pits, dumps, and proposed leach facility. Within the pit designs, ramps have been established for haul truck and equipment access. The in-pit ramps will only require a single berm. Ramps outside of the pit will require two safety berms. One-lane traffic ramps are also utilized near the bottom of pits where the strip ratio is minimal, and the traffic requirements are low.

The ramps and haul roads assume the use of CAT-793F haul trucks with an operating width of 8.56m. For two-way access the goal of the road design is to allow a running width of near 3.25 times the width of the trucks. MSHA regulations specify that safety berms be maintained at least ½ of the diameter of the tires of the haul trucks that will travel on roads. The ½ height of the CAT-793F haul trucks tires is 1.79 meters. An extra 10% was added to berm height design to ensure that all berms are a sufficient height.

Safety berms assume a slope of 1.5 horizontal to 1.0 vertical. Considering that ramps in the pit only need one berm, the road width of 34 meters was determined for two-lane traffic, which allows for 3.25 times the operating width of the haul trucks. Single-lane traffic roads are estimated to require 21 meters which allows 1.73 times the operating width of the CAT-793F haul trucks.

Roads outside of the pit will require two berms and widths are estimated to be 40 meters, allowing 3.23 times the width of the CAT-793F haul trucks.

Road designs are intended to have a maximum of 10% gradient, though some may exceed this for short distances around inside turns. Where switchbacks are utilized, the centerline gradient is reduced to about 8%. This keeps the inside gradient approximately 12%. Switchback designs have not added the detail for super elevation through the curves, but is it assumed that this will be done when they are constructed.

16.6 Pit Design

Pit designs were completed for DeLamar and Florida Mountain using Surpac™ software (version 6.7). Each of the designs utilize 6 meter benches with a catch bench of 9.6 meters wide installed every third bench, or 18 meters. The bench face angle used is 65°. The resulting inner-ramp slope is 45°.

The DeLamar and Florida Mountain pit designs are shown in the overall site plan in Figure 18.1.

Florida Mountain pit designs were completed using three pit phases. Phase one is mined as a smaller pit in the north east. Phase two is mined around phase one on the eastern side and then the final phase is mined to the west.

DeLamar pit designs utilize five pit phases to establish a mining sequence. The first mining at DeLamar would start in the Milestone pit and phase two mining will occur in the Sullivan pit. The remaining phases are in the DeLamar Main pit area, with phase three located in the northeast and phase five in the western portion. Phase 4 is a small round pit located on the southern end.

16.7 In-Pit Resources

In-pit mineral resources were estimated for each pit design. DeLamar in-pit resources are shown in Table 16.9. The DeLamar pits have a total of 32.2 million tonnes of waste associated with the material to be processed, and thus have an overall strip ratio of 0.74 tonnes of waste per tonne processed. The in-pit oxide and transition resources are reported using the 0.20g Au/t cutoff grade.

Table 16.9 DeLamar In-Pit Resources

	Units	Measured	Indicated	M&I	Inferred
Oxide	K Tonnes	570	20,094	20,664	3,857
	g Au/t	0.31	0.34	0.34	0.31
	K Ozs Au	6	218	224	39
	g Ag/t	21.31	17.60	17.70	13.17
	K Ozs Ag	390	11,367	11,757	1,633
Transition	K Tonnes	645	16,698	17,342	1,754
	g Au/t	0.39	0.37	0.37	0.32
	K Ozs Au	8	197	205	18
	g Ag/t	32.81	23.07	23.43	12.69
	K Ozs Ag	680	12,384	13,064	716
Total DeLamar	K Tonnes	1,214	36,792	38,006	5,611
In-Pit Resources	g Au/t	0.35	0.35	0.35	0.32
	K Ozs Au	14	415	429	57
	g Ag/t	27.42	20.08	20.31	13.02
	K Ozs Ag	1,070	23,751	24,821	2,349

Florida Mountain in-pit resources are shown in Table 16.10. The Florida Mountain mineral resources are associated with a total of 70.2 million tonnes of waste, resulting in a stripping ratio of 1.40 waste tonnes to processed tonnes. The in-pit oxide and transition resources are reported using a 0.20 g Au/t cutoff grade and unoxidized material uses a cutoff grade of 0.46 g Au/t.

Table 16.10 Florida Mountain In-Pit Resources

	Units	Measured	Indicated	M&I	Inferred
Oxide	K Tonnes	51	11,186	11,238	2,559
	g Au/t	0.76	0.42	0.42	0.32
	K Ozs Au	1	151	152	26
	g Ag/t	11.50	11.18	11.18	7.08
	K Ozs Ag	19	4,020	4,039	583
Transition	K Tonnes	1,074	28,538	29,612	2,418
	g Au/t	0.69	0.46	0.47	0.33
	K Ozs Au	24	420	443	26
	g Ag/t	17.49	11.77	11.98	7.18
	K Ozs Ag	604	10,797	11,402	558
UnOxidized	K Tonnes	95	3,925	4,019	287
	g Au/t	0.71	0.82	0.82	0.61
	K Ozs Au	2	104	106	6
	g Ag/t	13.13	17.54	17.44	13.56
	K Ozs Ag	40	2,213	2,253	125
Total Florida Mtn.	K Tonnes	1,220	43,649	44,870	5,263
In-Pit Resources	g Au/t	0.69	0.48	0.49	0.34
	K Ozs Au	27	674	701	57
	g Ag/t	16.90	12.14	12.27	7.48
	K Ozs Ag	663	17,031	17,694	1,266

The total in-pit resources considered for the PEA are shown in Table 16.11. Within the pits there is a total of 102.4 million tonnes of waste associated with the in-pit resources. This results in an overall project strip ratio of 1.09 tonnes of waste for each tonne of material processed.

Table 16.11 Total PEA In-Pit Resources

	Units	Measured	Indicated	M&I	Inferred
Oxide	K Tonnes	621	31,280	31,902	6,415
	g Au/t	0.35	0.37	0.37	0.31
	K Ozs Au	7	369	376	65
	g Ag/t	20.50	15.30	15.40	10.74
	K Ozs Ag	409	15,387	15,796	2,215
Transition	K Tonnes	1,719	45,236	46,955	4,172
	g Au/t	0.57	0.42	0.43	0.33
	K Ozs Au	32	617	648	44
	g Ag/t	23.24	15.94	16.21	9.50
	K Ozs Ag	1,284	23,181	24,465	1,274
UnOxidized	K Tonnes	95	3,925	4,019	287
	g Au/t	0.71	0.82	0.82	0.61
	K Ozs Au	2	104	106	6
	g Ag/t	13.13	17.54	17.44	13.56
	K Ozs Ag	40	2,213	2,253	125
Total PEA	K Tonnes	2,435	80,441	82,876	10,874
In-Pit Resources	g Au/t	0.52	0.42	0.42	0.33
	K Ozs Au	41	1,089	1,130	114
	g Ag/t	22.15	15.77	15.96	10.34
	K Ozs Ag	1,733	40,781	42,515	3,615

16.8 Dump and Leach Pad Design

Dump and leach pad designs were created for the PEA to contain the waste and heap-leach material mined. A 1.3 swell factor was assumed which provides for both swell when mined and compaction when placed into the facility. The total requirements for containment of waste and leach material are shown in Table 16.12.

Table 16.12 Waste and Leach Containment Requirements

	Waste Volumes (K Cu Meters) - w/ Swell				
	Ox_Wst	Mx_Wst	Su_Wst	Fill_Wst	Total
FlMtn_Ph_1	1,696	1,571	376	-	3,644
FlMtn_Ph_2	9,349	8,689	6,729	24	24,790
FlMtn_Ph_3	3,469	4,151	1,819	1,571	11,009
Total FlMnt	14,514	14,411	8,923	1,595	39,443
Del_Ph_1	2,202	102	142	-	2,445
Del_Ph_2	2,232	971	136	333	3,671
Del_Ph_3	2,804	1,218	1,021	1,042	6,084
Del_Ph_4	355	10	-	-	365
Del_Ph_5	2,651	952	625	3,054	7,283
Total Del	10,244	3,253	1,923	4,429	19,849
Total Project	24,758	17,664	10,847	6,024	59,293

Note: Cu m = cubic meters; Ox_Wst = oxide waste; Mx_Wst = transitional waste; Su_Wst = unoxidized waste; Fill_Wst = Fill Material; FlMtn = Florida Mountain; Del = DeLamar.
Swell factor of 1.3 was used

Dump designs were completed for both DeLamar and Florida Mountain. A total of three waste storage areas were designed for DeLamar due to the spread out nature of the DeLamar pits. The Florida Mountain waste storage facility is a single location.

Waste material may also be stored in the form of backfill where and when space is available. While this has not been assumed for the PEA, this is a potential opportunity for the project with future studies, particularly for backfilling of Florida Mountain phase 1 and DeLamar phase 3.

Waste dumps, the leach pad, and roads and facilities are shown in the general arrangement drawing in Figure 18.1.

16.9 Mine Production Schedule

Production scheduling was completed using Geovia's MineSched™ (version 9.1) software. Measured, Indicated, and Inferred resources inside of the pit designs previously discussed were used to schedule mine production.

The production schedule considers the processing of Florida Mountain oxide and transition material by crushing and heap leaching. Florida Mountain unoxidized material would be processed using flotation followed by cyanide leaching of the float concentrate. Processing of the DeLamar material will require crushing and agglomeration prior to heap leaching.

Monthly periods were used to create the production schedule with pre-stripping starting in Florida Mountain at month -6. Start of leach processing is in month 1, though a total of 468,000 tonnes of leach material is to be mined during preproduction. It is assumed that this material will be crushed and used as over liner on top of the leach pad liner. The maximum rate for leach processing will be 27,000 tonnes

per day or 9,720,000 tonnes per year. Note that during the first year, a total of 8,292,000 tonnes will be processed along with the material laid onto the pad during preproduction. This represents a ramp up to full processing.

Leaching starts with Florida Mountain material and DeLamar leach material is processed starting in year 5. Prior to that, the agglomeration circuit will be installed. The DeLamar leach material will be processed at the same rate as the Florida Mountain material.

Florida Mountain unoxidized material will be stockpiled until the flotation mill is constructed. The start of the 2,000 tonne per day mill will be in year 3 with 604,000 tonnes processed in that year, increasing to 72,0000 tonnes per year after that until the unoxidized material is exhausted.

The total mining rate would ramp up from an initial 2,000 tonnes per day to about 60,000 tonnes per day over a period of 15 months. A maximum of 90,000 tonnes per day is used in later years when the stripping requirement becomes more significant in Florida Mountain phase 3.

The yearly mining production for DeLamar and Florida Mountain is summarized in Table 16.13 and Table 16.14, respectively. Table 16.15 summarizes the yearly total mine production schedule.

Table 16.13 DeLamar Mine Production Schedule

		Units	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
DeLmar_Total	Leach Mined	K Tonnes	-	2,816	9,780	8,954	9,948	9,632	2,487	-	43,617
		g Au/t	-	0.35	0.36	0.33	0.35	0.35	0.35	-	0.35
		K Ozs Au	-	32	112	95	113	107	28	-	486
		g Ag/t	-	15.59	24.51	14.99	16.97	20.52	24.41	-	19.37
		K Ozs Ag	-	1,412	7,708	4,315	5,427	6,355	1,952	-	27,170
	Pit to Mill StkPl	K Tonnes	-	-	-	-	-	-	-	-	-
		g Au/t	-	-	-	-	-	-	-	-	-
		K Ozs Au	-	-	-	-	-	-	-	-	-
		g Ag/t	-	-	-	-	-	-	-	-	-
		K Ozs Ag	-	-	-	-	-	-	-	-	-
	Pit to Mill	K Tonnes	-	-	-	-	-	-	-	-	-
		g Au/t	-	-	-	-	-	-	-	-	-
		K Ozs Au	-	-	-	-	-	-	-	-	-
		g Ag/t	-	-	-	-	-	-	-	-	-
		K Ozs Ag	-	-	-	-	-	-	-	-	-
	Total Mined	K Tonnes	-	2,816	9,780	8,954	9,948	9,632	2,487	-	43,617
	Above COG	g Au/t	-	0.35	0.36	0.33	0.35	0.35	0.35	-	0.35
		K Ozs Au	-	32	112	95	113	107	28	-	486
		g Ag/t	-	15.59	24.51	14.99	16.97	20.52	24.41	-	19.37
		K Ozs Ag	-	1,412	7,708	4,315	5,427	6,355	1,952	-	27,170
	Ox_Wst	K Tonnes	-	4,479	3,298	4,056	2,676	1,891	537	-	16,938
	Mx_Wst	K Tonnes	-	384	1,654	1,393	1,343	675	264	-	5,712
	Su_Wst	K Tonnes	-	2	361	566	1,001	1,129	558	-	3,617
	Fill_Wst	K Tonnes	-	443	31	1,957	1,909	1,573	39	-	5,953
	Total Waste	K Tonnes	-	5,308	5,343	7,972	6,929	5,268	1,399	-	32,220
	Total Mined	K Tonnes	-	8,124	15,123	16,927	16,877	14,900	3,886	-	75,837
	Strip Ratio	W:O		1.88	0.55	0.89	0.70	0.55	0.56		0.74

COG = cutoff grade.

Table 16.14 Florida Mountain Mine Production Schedule

		Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Total
FlMnt_Total	Leach Mined	K Tonnes	468	8,615	9,662	10,330	8,905	7,009	837	-	45,827
		g Au/t	0.36	0.49	0.46	0.42	0.38	0.47	0.33	-	0.44
		K Ozs Au	5	135	143	140	109	105	9	-	647
		g Ag/t	7.76	13.12	9.37	9.72	9.72	15.25	17.57	-	11.25
		K Ozs Ag	117	3,634	2,910	3,229	2,783	3,436	473	-	16,582
	Pit to Mill StkPl	K Tonnes	-	153	309	618	1,463	671	123	-	3,336
		g Au/t	-	1.31	0.96	0.57	0.50	0.52	0.41	-	0.59
		K Ozs Au	-	6	10	11	23	11	2	-	63
		g Ag/t	-	9.89	13.33	9.82	14.36	12.60	18.32	-	13.01
		K Ozs Ag	-	49	132	195	676	272	72	-	1,395
	Pit to Mill	K Tonnes	-	-	-	191	546	164	69	-	970
		g Au/t	-	-	-	1.56	1.60	1.29	1.49	-	1.53
		K Ozs Au	-	-	-	10	28	7	3	-	48
		g Ag/t	-	-	-	27.59	26.68	36.09	69.95	-	31.51
		K Ozs Ag	-	-	-	169	469	191	154	-	983
	Total Mined	K Tonnes	468	8,768	9,971	11,139	10,915	7,844	1,029	-	50,133
	Above COG	g Au/t	0.36	0.50	0.48	0.45	0.46	0.49	0.42	-	0.47
		K Ozs Au	5	142	152	161	161	123	14	-	758
		g Ag/t	7.76	13.06	9.49	10.03	11.19	15.46	21.15	-	11.76
		K Ozs Ag	117	3,682	3,042	3,593	3,927	3,898	699	-	18,960
	Ox_Wst	K Tonnes	768	5,043	6,077	5,182	5,686	1,791	310	-	24,857
	Mx_Wst	K Tonnes	135	3,115	5,374	6,970	6,380	3,795	258	-	26,026
	Su_Wst	K Tonnes	-	534	445	5,844	7,182	2,740	417	-	17,160
	Un_Wst	K Tonnes	-	-	-	-	-	-	-	-	-
	Fill_Wst	K Tonnes	-	-	33	-	2,071	73	-	-	2,177
	Total Waste	K Tonnes	903	8,691	11,929	17,996	21,318	8,399	984	-	70,221
	Total Mined	K Tonnes	1,371	17,459	21,900	29,135	32,233	16,243	2,013	-	120,354
	Strip Ratio	W:O	1.93	0.99	1.20	1.62	1.95	1.07	0.96		1.40

Table 16.15 Total PEA Mine Production Schedule

		Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Total Project	Leach Mined	K Tonnes	468	8,615	9,662	10,330	8,905	9,825	10,617	8,954	9,948	9,632	2,487	-	89,444
		g Au/t	0.36	0.49	0.46	0.42	0.38	0.43	0.35	0.33	0.35	0.35	0.35	-	0.39
		K Ozs Au	5	135	143	140	109	137	121	95	113	107	28	-	1,132
		g Ag/t	7.76	13.12	9.37	9.72	9.72	15.35	23.97	14.99	16.97	20.52	24.41	-	15.21
		K Ozs Ag	117	3,634	2,910	3,229	2,783	4,848	8,181	4,315	5,427	6,355	1,952	-	43,751
	Pit to Mill StkPl	K Tonnes	-	153	309	618	1,463	671	123	-	-	-	-	-	3,336
		g Au/t	-	1.31	0.96	0.57	0.50	0.52	0.41	-	-	-	-	-	0.59
		K Ozs Au	-	6	10	11	23	11	2	-	-	-	-	-	63
		g Ag/t	-	9.89	13.33	9.82	14.36	12.60	18.32	-	-	-	-	-	13.01
		K Ozs Ag	-	49	132	195	676	272	72	-	-	-	-	-	1,395
	Pit to Mill	K Tonnes	-	-	-	191	546	164	69	-	-	-	-	-	970
		g Au/t	-	-	-	1.56	1.60	1.29	1.49	-	-	-	-	-	1.53
		K Ozs Au	-	-	-	10	28	7	3	-	-	-	-	-	48
		g Ag/t	-	-	-	27.59	26.68	36.09	69.95	-	-	-	-	-	31.51
		K Ozs Ag	-	-	-	169	469	191	154	-	-	-	-	-	983
	Total Mined	K Tonnes	468	8,768	9,971	11,139	10,915	10,660	10,809	8,954	9,948	9,632	2,487	-	93,750
	Above COG	g Au/t	0.36	0.50	0.48	0.45	0.46	0.45	0.36	0.33	0.35	0.35	0.35	-	0.41
		K Ozs Au	5	142	152	161	161	155	126	95	113	107	28	-	1,244
		g Ag/t	7.76	13.06	9.49	10.03	11.19	15.49	24.19	14.99	16.97	20.52	24.41	-	15.30
		K Ozs Ag	117	3,682	3,042	3,593	3,927	5,310	8,408	4,315	5,427	6,355	1,952	-	46,130
	Ox_Wst	K Tonnes	768	5,043	6,077	5,182	5,686	6,270	3,608	4,056	2,676	1,891	537	-	41,795
	Mx_Wst	K Tonnes	135	3,115	5,374	6,970	6,380	4,179	1,911	1,393	1,343	675	264	-	31,739
	Su_Wst	K Tonnes	-	534	445	5,844	7,182	2,742	778	566	1,001	1,129	558	-	20,777
	Fill_Wst	K Tonnes	-	-	33	-	2,071	516	31	1,957	1,909	1,573	39	-	8,129
	Total Waste	K Tonnes	903	8,691	11,929	17,996	21,318	13,707	6,328	7,972	6,929	5,268	1,399	-	102,440
	Total Mined	K Tonnes	1,371	17,459	21,900	29,135	32,233	24,367	17,136	16,927	16,877	14,900	3,886	-	196,190
	Strip Ratio	W:O	1.93	0.99	1.20	1.62	1.95	1.29	0.59	0.89	0.70	0.55	0.56		1.09

The leach process production schedule was created by MDA based on the mine production schedule, metal recoveries and lag times. The recoveries used to estimate recoverable gold are those shown in Table 16.2. The lag time is generated by estimating the extraction of the recoverable ounces on a month by month basis after placement of material. The assumed rate of extraction of the recoverable ounces by month for leach material is as follows:

- Month placed = 0% of recoverable ounces;
- First month after placement = 60% of recoverable ounces;
- Second month after placement = 20% of recoverable ounces;
- Third month after placement = 10% of recoverable ounces;
- Fourth month after placement = 5% of recoverable ounces;
- Fifth month after placement = 3% of recoverable ounces; and
- Sixth month after placement = 2% of recoverable ounces.

Lag times for material placed during preproduction as part of the over liner is delayed to start at month one.

The Florida Mountain mill production does not assume any lag time.

Table 16.16 shows the yearly process production summary by process type. Rows labeled "K Au Rec" shows the thousands of recoverable ounces of gold and rows labeled "K Au Prod" are the thousands of ounces of gold produced.

The PEA total life-of-mine ("LOM") gold production is estimated to be 1,036,000 ounces, with a LOM average recovery of 83%. Silver production is estimated to be 16,686,000 ounces, with an average LOM recovery of 36%.

Table 16.16 PEA Process Production Schedule

	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Florida Mtn Leach	K Tonnes	468	8,292	9,720	9,747	9,720	7,043	837	-	-	-	-	-	45,827
	g Au/t	0.36	0.50	0.46	0.43	0.37	0.47	0.33	-	-	-	-	-	0.44
	K Ozs Au	5	133	143	135	116	106	9	-	-	-	-	-	647
	K Ozs Au Rec	5	115	124	116	101	91	8	-	-	-	-	-	559
	K Ozs Au Prod	-	97	130	115	101	104	12	-	-	-	-	-	559
	g Ag/t	7.76	13.31	9.41	9.86	9.49	15.21	17.57	-	-	-	-	-	11.25
	K Ozs Ag	117	3,549	2,941	3,091	2,966	3,445	473	-	-	-	-	-	16,582
	K Ozs Ag Rec	47	1,420	1,176	1,236	1,187	1,378	189	-	-	-	-	-	6,633
	K Ozs Ag Prod	-	1,235	1,268	1,230	1,139	1,477	283	-	-	-	-	-	6,633
DeLamar Leach	K Tonnes	-	-	-	-	-	2,677	8,869	9,747	9,720	9,720	2,885	-	43,617
	g Au/t	-	-	-	-	-	0.35	0.36	0.32	0.35	0.35	0.34	-	0.35
	K Ozs Au	-	-	-	-	-	30	104	101	111	108	31	-	486
	K Ozs Au Rec	-	-	-	-	-	24	81	79	85	84	24	-	377
	K Ozs Au Prod	-	-	-	-	-	15	78	78	84	86	36	-	377
	g Ag/t	-	-	-	-	-	15.77	25.46	15.07	16.99	20.33	23.40	-	19.37
	K Ozs Ag	-	-	-	-	-	1,357	7,260	4,721	5,309	6,352	2,170	-	27,170
	K Ozs Ag Rec	-	-	-	-	-	407	2,178	1,416	1,593	1,906	651	-	8,151
	K Ozs Ag Prod	-	-	-	-	-	247	2,001	1,538	1,552	1,820	994	-	8,151
Total Leach	K Tonnes	468	8,292	9,720	9,747	9,720	9,720	9,706	9,747	9,720	9,720	2,885	-	89,444
	g Au/t	0.36	0.50	0.46	0.43	0.37	0.44	0.36	0.32	0.35	0.35	0.34	-	0.39
	K Ozs Au	5	133	143	135	116	136	113	101	111	108	31	-	1,132
	K Ozs Au Rec	5	115	124	116	101	115	88	79	85	84	24	-	936
	K Ozs Au Prod	-	97	130	115	101	119	90	78	84	86	36	-	936
	g Ag/t	7.76	13.31	9.41	9.86	9.49	15.37	24.78	15.07	16.99	20.33	23.40	-	15.21
	K Ozs Ag	117	3,549	2,941	3,091	2,966	4,802	7,733	4,721	5,309	6,352	2,170	-	43,751
	K Ozs Ag Rec	47	1,420	1,176	1,236	1,187	1,785	2,367	1,416	1,593	1,906	651	-	14,784
	K Ozs Ag Prod	-	1,235	1,268	1,230	1,139	1,724	2,285	1,538	1,552	1,820	994	-	14,784
Florida Mtn Mill	K Tonnes	-	-	-	604	720	720	720	722	720	100	-	-	4,306
	g Au/t	-	-	-	1.27	1.38	0.69	0.60	0.51	0.51	0.51	-	-	0.80
	K Ozs Au	-	-	-	25	32	16	14	12	12	2	-	-	111
	K Ozs Au Rec	-	-	-	22	29	14	13	11	11	1	-	-	100
	K Ozs Au Prod	-	-	-	22	29	14	13	11	11	1	-	-	100
	g Ag/t	-	-	-	17.30	23.15	18.22	18.57	13.21	13.21	13.21	-	-	17.18
	K Ozs Ag	-	-	-	336	536	422	430	307	306	43	-	-	2,378
	K Ozs Ag Rec	-	-	-	269	429	337	344	245	245	34	-	-	1,903
	K Ozs Ag Prod	-	-	-	269	429	337	344	245	245	34	-	-	1,903
Total Project	K Tonnes	468	8,292	9,720	10,350	10,440	10,440	10,426	10,469	10,440	9,820	2,885	-	93,750
	g Au/t	0.36	0.50	0.46	0.48	0.44	0.45	0.38	0.34	0.36	0.35	0.34	-	0.41
	K Ozs Au	5	133	143	159	148	152	127	113	122	110	31	-	1,244
	K Ozs Au Rec	5	115	124	138	129	129	101	89	96	86	24	-	1,036
	K Ozs Au Prod	-	97	130	137	130	133	102	89	95	87	36	-	1,036
	g Ag/t	7.76	13.31	9.41	10.30	10.43	15.56	24.35	14.94	16.73	20.25	23.40	-	15.30
	K Ozs Ag	117	3,549	2,941	3,426	3,502	5,224	8,163	5,028	5,615	6,394	2,170	-	46,130
	K Ozs Ag Rec	47	1,420	1,176	1,505	1,615	2,122	2,711	1,662	1,837	1,940	651	-	16,686
	K Ozs Ag Prod	-	1,235	1,268	1,498	1,568	2,061	2,629	1,783	1,796	1,854	994	-	16,686

Although not yet designed, stockpiles are anticipated to be located near the crushing facility. The stockpiles would be used to store low-grade material longer term as well as some higher grade material during initial mining.

Table 16.17, Table 16.18, and Table 16.19 show stockpile balance sheets for the Florida Mountain leach, DeLamar leach, and Florida Mountain mill stockpiles.

Table 16.17 Florida Mountain Leach Stockpile Balance

Florida Mountian Leach	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11
Added to StkPl	K Tonnes	407	5,721	6,811	7,695	6,665	4,574	659	-	-	-	-	-
	g Au/t	0.31	0.27	0.28	0.27	0.26	0.26	0.23	-	-	-	-	-
	K Ozs Au	4	50	60	66	56	38	5	-	-	-	-	-
	g Ag/t	7.20	7.91	6.35	7.05	7.92	10.92	13.88	-	-	-	-	-
	K Ozs Ag	94	1,455	1,391	1,743	1,698	1,606	294	-	-	-	-	-
Removed from StkPl	K Tonnes	407	5,398	6,869	7,111	7,480	4,608	659	-	-	-	-	-
	g Au/t	0.31	0.27	0.28	0.27	0.26	0.26	0.23	-	-	-	-	-
	K Ozs Au	4	47	61	61	63	38	5	-	-	-	-	-
	g Ag/t	7.20	7.90	6.44	7.02	7.82	10.90	13.88	-	-	-	-	-
	K Ozs Ag	94	1,371	1,422	1,605	1,881	1,615	294	-	-	-	-	-
StkPl Balance	K Tonnes	-	323	265	848	34	-	-	-	-	-	-	-
	g Au/t	-	0.27	0.28	0.27	0.28	-	-	-	-	-	-	-
	K Ozs Au	-	3	2	7	0	-	-	-	-	-	-	-
	g Ag/t	-	8.14	6.28	7.04	8.05	-	-	-	-	-	-	-
	K Ozs Ag	-	84	53	192	9	-	-	-	-	-	-	-

Table 16.18 DeLamar Leach Stockpile Balance

DeLamar Leach	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11
Added to StkPl	K Tonnes	-	-	-	-	-	1,975	6,385	6,894	7,248	6,527	1,659	-
	g Au/t	-	-	-	-	-	0.25	0.26	0.27	0.27	0.26	0.24	-
	K Ozs Au	-	-	-	-	-	16	54	59	63	54	13	-
	g Ag/t	-	-	-	-	-	11.72	14.52	12.01	11.92	14.64	15.60	-
	K Ozs Ag	-	-	-	-	-	744	2,980	2,663	2,778	3,072	832	-
Removed from StkPl	K Tonnes	-	-	-	-	-	1,836	5,474	7,686	7,020	6,615	2,057	-
	g Au/t	-	-	-	-	-	0.25	0.26	0.27	0.27	0.26	0.25	-
	K Ozs Au	-	-	-	-	-	15	46	66	61	55	16	-
	g Ag/t	-	-	-	-	-	11.69	14.39	12.42	11.79	14.43	15.88	-
	K Ozs Ag	-	-	-	-	-	690	2,532	3,069	2,660	3,068	1,050	-
StkPl Balance	K Tonnes	-	-	-	-	-	139	1,051	258	486	398	-	-
	g Au/t	-	-	-	-	-	0.26	0.27	0.27	0.27	0.26	-	-
	K Ozs Au	-	-	-	-	-	1	9	2	4	3	-	-
	g Ag/t	-	-	-	-	-	12.18	14.89	11.70	13.75	17.05	-	-
	K Ozs Ag	-	-	-	-	-	54	503	97	215	218	-	-

Table 16.19 Florida Mountain Mill Stockpile Balance

Florida Mountian Mill	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11
Added to StkPl	K Tonnes	-	153	309	618	1,463	671	123	-	-	-	-	-
	g Au/t	-	1.32	0.96	0.57	0.50	0.52	0.41	-	-	-	-	-
	K Ozs Au	-	6	10	11	23	11	2	-	-	-	-	-
	g Ag/t	-	9.89	13.33	9.82	14.36	12.60	18.32	-	-	-	-	-
	K Ozs Ag	-	49	132	195	676	272	72	-	-	-	-	-
Removed from StkPl	K Tonnes	-	-	-	413	174	556	651	722	720	100	-	-
	g Au/t	-	-	-	1.14	0.65	0.51	0.51	0.51	0.51	0.51	-	-
	K Ozs Au	-	-	-	15	4	9	11	12	12	2	-	-
	g Ag/t	-	-	-	12.55	12.04	12.93	13.16	13.21	13.21	13.21	-	-
	K Ozs Ag	-	-	-	167	67	231	276	307	306	43	-	-
StkPl Balance	K Tonnes	-	153	461	666	1,956	2,071	1,542	820	100	-	-	-
	g Au/t	-	1.32	1.08	0.58	0.51	0.51	0.51	0.51	0.51	-	-	-
	K Ozs Au	-	6	16	12	32	34	25	13	2	-	-	-
	g Ag/t	-	9.89	12.19	9.76	13.00	12.89	13.21	13.21	13.21	-	-	-
	K Ozs Ag	-	49	181	209	818	858	655	348	43	-	-	-

16.10 Equipment Requirements

The PEA has assumed owner mining over the more expensive contract mining. The production schedule was used along with additional efficiency factors, cycle times, and productivity rates to develop the first principle hours required for primary mining equipment to achieve the production schedule. Primary mining equipment includes drills, loaders, hydraulic shovels, and haul trucks.

Support, blasting, and mine maintenance equipment would be required in addition to the primary mining equipment. Table 16.20 shows the yearly equipment requirements.

Table 16.20 PEA Yearly Mine Equipment Requirements

Primary Equipment	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Max
Production Drills	#	1	3	3	4	4	4	3	3	3	3	2	-	4
Pioneering Drills	#	1	-	-	-	-	2	-	-	-	-	-	-	2
Loader	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Hydraulic Shovel	#	1	1	2	2	2	2	2	1	1	1	1	-	2
Haul Trucks	#	3	10	11	15	15	15	11	8	8	8	8	-	15
Support Equipment														
D10 Type Dozer	#	1	2	2	2	2	2	2	2	2	2	2	-	2
D9 Type Dozer	#	1	1	1	1	1	1	1	1	1	1	1	-	1
D8 Type Dozer	#	-	-	-	-	-	-	-	-	-	-	-	-	-
Motor Grader (18')	#	2	2	2	2	2	2	2	2	2	2	2	-	2
Water Truck - 20,000 gal	#	1	2	2	2	2	2	2	2	2	2	2	-	2
Pit Pumps	#	1	2	2	2	2	2	2	2	2	2	2	-	2
50 Ton Crane	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Flat Bed Truck	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Blasting														
Skid Loader	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Explosives Truck	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Mine Maintenance														
Lube/Fuel Truck	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Mechanic/Service Truck	#	1	2	2	2	2	2	2	2	2	2	2	-	2
Tire Truck	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Other Mine Equipment														
Light Plants	#	4	4	4	4	4	4	4	4	4	4	4	-	4

Note that the pit designs discussed earlier in this section anticipated use of larger trucks. However the final production and cost evaluation uses CAT-785 136-tonne capacity haul trucks.

The mine is anticipated to operate 24 hours per day, utilizing four crews of workers each working four days on and four days off. It is anticipated that these crews would rotate between day shift and night shift. The daily shift schedule would be 12 hours per day reduced to account for standby time including startup/shutdown, lunch, breaks, and operational delays totaling 3 hours per day. This allows for 21 work hours in each day or 87.5% schedule efficiency. The estimated schedule efficiency is shown in Table 16.21.

Table 16.21 Schedule Efficiency

	Units	Value
Shifts per Day	shift/day	2
Hours per Shift	hr/shift	12
Theoretical Hours per Day	hrs/day	24
Shift Startup / Shutdown	hrs/shift	0.5
Lunch	hrs/shift	0.5
Breaks	hrs/shift	0.25
Operational Standby	hrs/shift	0.25
Total Standby / shift	hrs/shift	1.50
Total Standby / day	hrs/day	3.00
Available Work Hours	hrs/day	21.00
Schedule Efficiency	%	87.5%

Pioneer drills would be smaller air-track drills with contained cabs and the production drills are anticipated to be 45,000lb-pulldown, track-mounted, rotary blast-hole drills. An 83% efficiency factor was used for pioneer drilling and 85% efficiency was used for production and controlled blast hole drilling. Penetration rates of 31.6, 33.1, and 36.5 meters per hour were used along with 2.8, 2.8, and 3.0 minutes per hole of non-drilling times for production, trim-rows, and pioneer drilling, respectively.

Based on the parameters used, two pioneer drills and four production drills are estimated to be needed. It is assumed that these drills will last through the LOM with an availability that is assumed to be 85% for the life of the drill.

Loading equipment is anticipated to include one CAT-994 type loader and two CAT 6040B type hydraulic shovels. The CAT-994 loading theoretical productivity was estimated to be 2,345 tonnes per hour, or 1,950 tonnes per hour at an operating efficiency of 83%. The loader is primarily used for back-up mining production and re-handle of material from stockpiles. The assumed availability starts at 90% and is reduced 1% per year until it reaches 85%, and then is held constant through the life of the shovels. No replacement loaders were assumed. The overall use of available hours is 43%.

Two CAT-6040B style shovels are used as the primary loading tool. The initial shovel starts operating in month -6 and the second shovel starts working in month 17. The theoretical productivity was estimated to be 3,326 tonnes per hour, or 2,760 tonnes per hour after applying 83% efficiency. As with

the loader, the assumed availability starts at 90% and declines at 1% per year to a low of 85%, and then remains the same through the LOM. The overall use of operating hours is 63%.

Haul trucks are assumed to be CAT-785 type 136-tonne capacity rigid frame trucks. Cycle times were assumed based on centerline distances draw from the pits to destinations. Average cycle times for each pit phase were used. The productivity was based on the cycle times along with load and spot time, spot and dump time, and 83% efficiency. The load time depended on whether the truck was loaded by a loader or shovel. The loader time used was 3.73 minutes and the shovel time used was 2.67 minutes. Spot time at the loader or shovel was 0.50 minutes and the spot and dump time was a combined 1.50 minutes. A capacity of 131 tonnes per load was used as dry tonnage to reflect the dry densities in the resource block model. The number of trucks was calculated to increase over time due to farther haulage with some pit phases. A total of 15 haul trucks are purchased to maintain the production schedule. This assumes a 1% per year declining availability from 90% down to 85%.

16.11 Personnel Requirements

Table 16.22 shows the estimated mine operations personnel requirements (full mine site personnel requirements are shown in Table 18.1). This is based on the number of people that will be required to operate, supervise, maintain, and plan for operations to achieve the production schedule. An average of 107 operating personnel will be required for mining operations. The peak mining personnel requirement is 217 people on an annual basis.

Table 16.22 PEA Mining Personnel Requirements

Mining General Personnel	**Units**	**Pre-Prod**	**Yr_1**	**Yr_2**	**Yr_3**	**Yr_4**	**Yr_5**	**Yr_6**	**Yr_7**	**Yr_8**	**Yr_9**	**Yr_10**	**Yr_11**	**Maximum**
Mine Superintendent	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Mine General Foreman	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Mine Foremen	#	4	4	4	4	4	4	4	4	4	4	4	-	4
Chief Mine Engineer	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Mine Engineer	#	2	2	2	2	2	2	2	2	2	2	2	-	2
Chief Surveyor	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Surveyor	#	3	3	3	3	3	3	3	3	3	3	3	-	3
Chief Geologist	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Ore Control Geologist	#	4	4	4	4	4	4	4	4	4	4	4	-	4
Samplers	#	4	4	4	4	4	4	4	4	4	4	4	-	4
Total Mine General	#	22	22	22	22	22	22	22	22	22	22	22	-	22
Mine Operations Hourly Personnel														
Operators														
Blasters	#	2	2	2	2	2	2	2	2	2	2	2	-	2
Blaster's Helpers	#	2	2	2	2	2	2	2	2	2	2	2	-	2
Drill Operators	#	8	12	12	16	16	20	12	12	12	12	8	-	20
Loader Operators	#	4	6	10	10	10	10	8	6	6	6	6	-	10
Haul Truck Operators	#	12	40	44	60	60	60	44	32	32	32	32	-	60
Support Equipment Operators	#	12	17	17	17	17	17	17	17	17	17	17	-	17
General Mine Labors	#	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Operators	#	40	79	87	107	107	111	85	71	71	71	67	-	111
Mechanics														
Mechanics - Drilling	#	4	6	6	8	8	10	6	6	6	6	4	-	10
Mechanics - Loading	#	4	6	6	8	8	10	6	6	6	6	4	-	10
Mechanics - Haulage	#	6	20	22	30	30	30	22	16	16	16	16	-	30
Mechanics - Support	#	6	9	9	9	9	9	9	9	9	9	9	-	9
Total Mechanics	#	20	41	43	55	55	59	43	37	37	37	33	-	59
Maintenance														
Maintenance Superintendent	#	1	1	1	1	1	1	1	1	1	1	1	-	1
Maintenance Foreman	#	4	4	4	4	4	4	4	4	4	4	4	-	4
Maintenance Planners	#	2	2	2	2	2	2	2	2	2	2	2	-	2
Light Vehicle Mechanic	#	2	2	2	2	2	2	2	2	2	2	2	-	2
Welder	#	4	4	4	4	4	4	4	4	4	4	4	-	4
Servicemen	#	4	4	4	4	4	4	4	4	4	4	4	-	4
Tireman	#	4	4	4	4	4	4	4	4	4	4	4	-	4
Maintenance Labor	#	4	4	4	4	4	4	4	4	4	4	4	-	4
Total Maintenance	#	25	25	25	25	25	25	25	25	25	25	25	-	25
Total Mine Operations														
Total	#	107	167	177	209	209	217	175	155	155	155	147	-	217

17.0 RECOVERY METHODS

Mr. Jeff Woods of Woods Process Services in Denver, Colorado, prepared this section. The term "ore" is used in this section only in a metallurgical sense, to indicate mineralized material processed.

The proposed processing facilities for the DeLamar project will be developed in three stages corresponding to the mining sequence of the Florida Mountain and DeLamar deposits. The primary processing methods include heap leaching of secondary and tertiary crushed ore for oxide and transitional ore types, followed by Merrill-Crowe recovery of precious metals. For non-oxide ore types, the proposed process focuses on flotation and concentrate leaching followed by counter current decantation ("CCD") with precious-metal recovery using Merrill-Crowe zinc precipitation. The development phases are summarized in Table 17.1 and discussed in greater in the following text.

Table 17.1 DeLamar Process Development Phase

Phase	Deposit/Primary Ore Type	Processing Flowsheet	Projected Commissioning Year
1	Florida Mountain Oxide and Transitional	30 ktpd Two Stage Crushing, Heap Leach with Merrill Crowe recovery.	0
2	Florida Mountain Unoxidized	2 ktpd Concentrator with concentrate leach and CCD	2
3	DeLamar Oxide and Transitional	30 ktpd Three Stage Crushing, Agglomeration, Heap Leach with Merrill Crowe recovery.	5

17.1 Summary Process Design Criteria

Table 17.2 lists the preliminary design process for each of the process facilities and is grouped by ore type and deposit. It should be noted that there are several processing circuits which have not been optimized and require additional test work to develop completely.

Table 17.2 DeLamar Project Process Design Criteria

Parameter	DeLamar	Florida Mountain
Heap Leach - Oxide	Crush/Agglomerate/Heap Leach	Crush/Heap Leach
Crushing Circuit Configuration	Three Stage Crushing with Agglomeration	Two Stage Crushing
Crush Size	80%-13mm	80%-38mm
Heap Stacking Method	Overland Conveyor with Radial Stacker	Overland Conveyor with Radial Stacker
Leach Cycle, days	120	120
Lift Height, m	8	8
Solution Application Rate, L/hr/m^2	10	10
Nominal Barren Solution Flow, m^3/hr	1,250	1,250
Solution Application Method	Drip	Drip
Precious Metal Recovery Method	Merrill-Crowe, Zinc Precipitation	Merrill-Crowe, Zinc Precipitation

Parameter	DeLamar	Florida Mountain
Nominal PLS Flow Rate to Plant, m^3/hr	1,175	1,175
Au Recovery	80%	90%
Ag Recovery	30%	40%
NaCN, kg/t	0.3	0.4
Lime, kg/t	---	1.0
Cement, kg/t	3.0	---
Heap Leach - Transitional	Crush/Agglomerate/Heap Leach	Crush/Heap Leach
Crushing Circuit Configuration	Three Stage Crushing; third stage closed circuit with Agglomeration	Two Stage Crushing in open circuit
Crush Size	80%-13mm	80%-38mm
Leach Cycle, days	120	120
Solution Application Rate, L/hr/m^2	10	10
Nominal Barren Solution Flow, m^3/hr	1,250	1,250
Solution Application Method	Drip	Drip
Precious Metal Recovery Method	Merrill-Crowe, Zinc Precipitation	Merrill-Crowe, Zinc Precipitation
Nominal PLS Flow Rate to Plant, m^3/hr	1,175	1,175
Au Recovery	75%	85%
Ag Recovery	30%	40%
NaCN, kg/t	0.4	0.4
Lime, kg/t	---	1.0
Cement, kg/t	3.0	---
Milling - Unoxidized Material Type		
Grind/Gravity/Flotation with Concentrate Regrind/Leach	Not considered for PEA	Grind/Gravity/Flotation/Conc. Leach
Crushing Circuit Configuration		Three Stage Crushing; third stage closed circuit
Crush Size		80% -6mm
Primary Grind Size		80%-212µm
Gravity Concentrator Type		Centrifugal
Flotation Cell Type		Tank Cell
Flotation Circuit Configuration		Rougher-Rougher Scavenger-1st Cleaner-2nd Cleaner
Rougher Flotation Retention Time, min		40
Rougher Scavenger Retention Time, min		20
First Cleaner Retention Time, min		20
Second Cleaner Retention Time, min		20
Flotation Tailings Handling Method		Thicken, Filter, Dry stack
Concentrate Mass (for regrind) Percent of Total Feed		5%
Regrind Size		95%-37µm

Parameter	DeLamar	Florida Mountain
Potassium Amyl Xanthate (PAX), kg/t		0.025
AERO 208 (Dithiophosphate), kg/t		0.050
AEROFROTH 65 (Polyglycol Frother), kg/t		0.1
Concentrate Leach Solids Density, % wt:wt		0.5
Concentrate Leach Retention Time		24 hrs.
NaCN Consumption, kg/t		0.2
Lime Consumption, kg/t		0.2
Au Recovery		90%
Ag Recovery		80%
Leach Tailings Handling Method		Thicken, Add to Heap Leach Feed

17.2 Process Descriptions

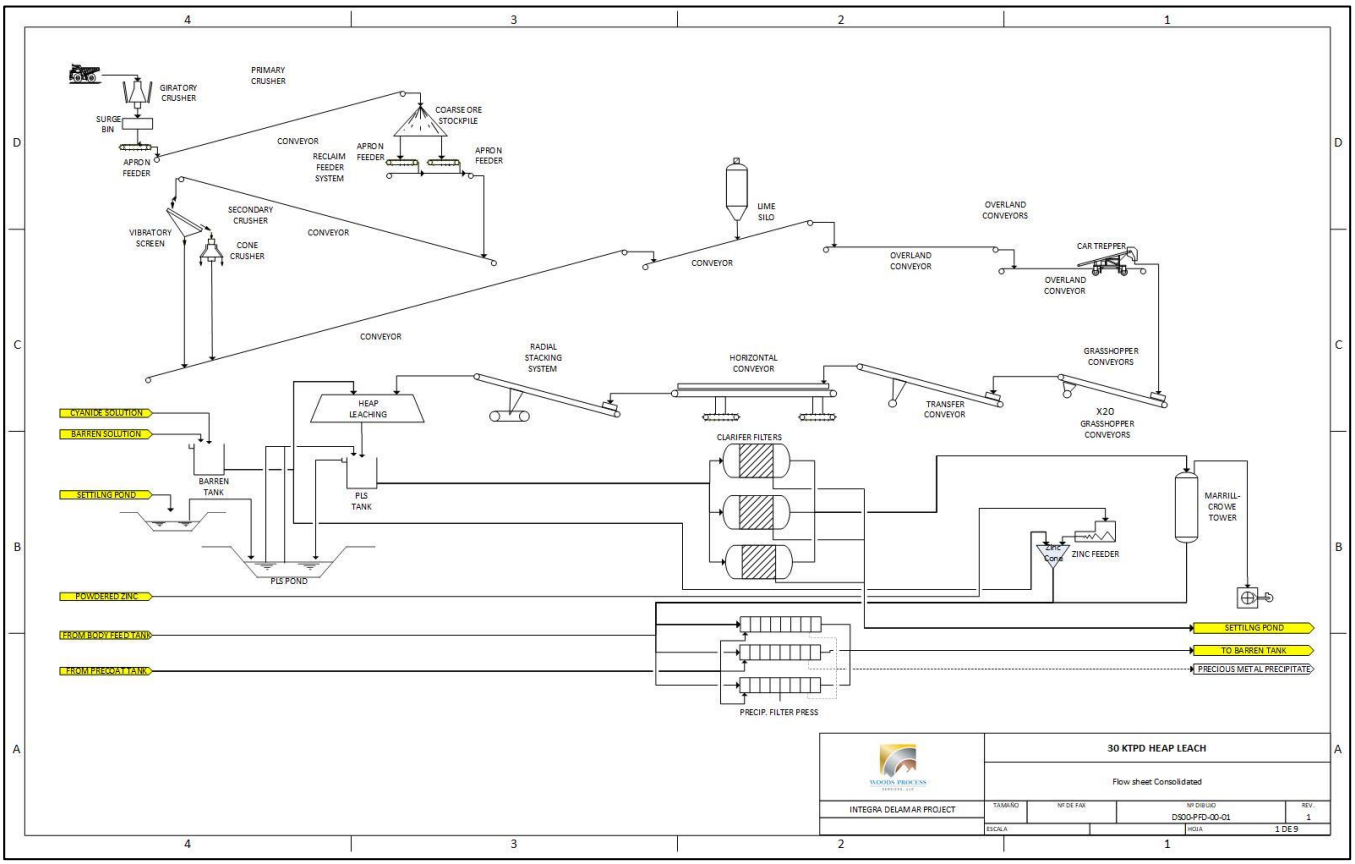
The following subsections present the proposed process flow sheets and a brief process description for each of the process facilities. They are arranged in chronological order corresponding to construction and commissioning sequence.

17.2.1 Phase I: Florida Mountain 30 ktpd Heap Leach

The proposed initial processing facility is designed to process the Florida Mountain oxide and transitional ore types using conventional heap leaching for the extraction of precious metals and solution recovery using Merrill-Crowe zinc precipitation. The proposed simplified Florida Mountain process flow sheet is shown in Figure 17.1.

ROM material will be crushed to a nominal 100 millimeter size using a gyratory crusher for primary crushing and a secondary cone crusher operating in open circuit. The crushing plant will operate 20 hours per day for two 12-hour shifts, seven days a week, to supply the crushed ore stockpile at a nominal production rate of 1,659 tph. The crushed ore stockpile will have enough capacity to feed the downstream crushing operations to provide ore to heap leaching stacking circuit, which will operate with two, twelve-hour shifts to continuously operate 24 hours/day and 7 days a week. The crushed ore stockpile will provide buffering capacity to minimize production loss during primary crusher maintenance and stacking conveyor moves.

Figure 17.1 Florida Mountain 30 ktpd Heap Leach Process Flow Sheet.



Crushed ore will be reclaimed from the crushed ore stockpile via two apron feeders and ore reporting to the secondary vibratory screen. The screen undersize reports to the secondary product conveyor. The secondary screen oversize reports to the secondary cone crusher to produce a final product with an 80 percent passing size of 38mm. The secondary cone crusher product is discharged to the secondary product conveyor and then onto the final product conveyor where lime is added for pH control on the heap at a rate of 1.0 kg/t.

The final crusher product mixed with lime reports to the first of two overland conveyors. The second overland conveyor is equipped with a moveable tripper conveyor which can be moved along the second overland conveyor to transfer crushed ore to the heap stacking system. The initial stacking system will be comprised of 20 grasshopper conveyors, and an inclined conveyor to transfer ore the horizontal-radial stacking conveyor. The horizontal-radial stacking conveyor is coupled so that it can be moved in a retreat stacking mode without shutting down the system. The proposed stacking lift height is 8.0 m resulting in a nominal daily heap surface area increase of 3,200 m^2 per day.

After stacking, the piping heads and drip irrigation lines will be added to the heap surface. Sodium cyanide solution will be applied to the heap surface via the header/drip system at a proposed application rate of 10 L/hr/m^2 for the preliminary leach cycle of 120 days. The cyanide solution at a nominal flow

rate of 1,100 m^3 /hr applied to the heap surface will percolate though the heap, being collected on the impervious leach pad liner, flowing out of the heap and being collected in the pregnant leach solution ("PLS") recovery system. The PLS reports to the PLS tank before being pumped to the Merrill-Crowe circuit for metal recovery. It is planed that the heap-leach solution application rate will be adjusted during the leach cycle to maximize the PLS gold and silver content.

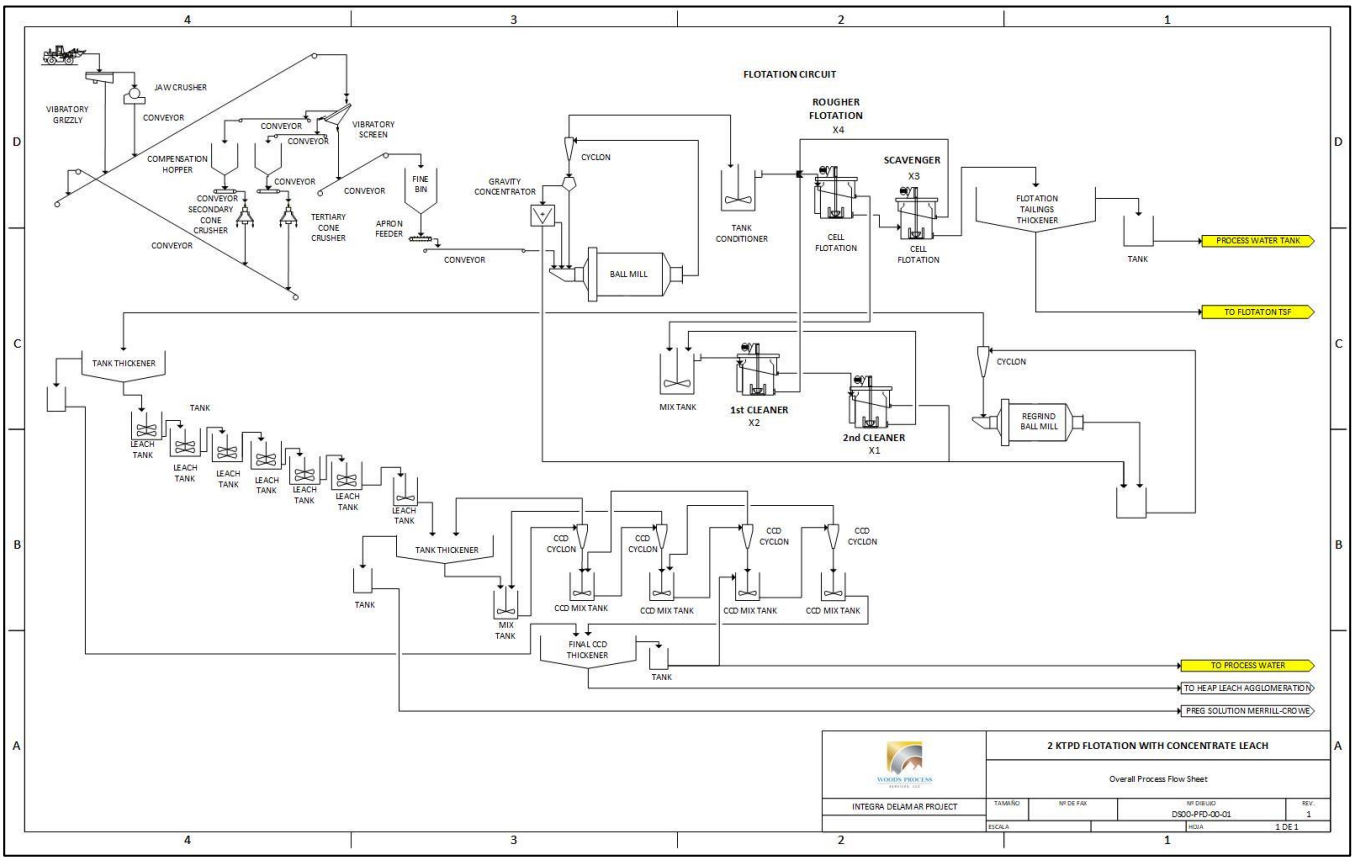
The PLS reporting the PLS tank will be pumped to the Merrill-Crowe circuit, first being clarified in the clarifier filters and then reporting to the deaeration tower where dissolved oxygen is removed from the PLS. The clarifier filters will be a set of three filters operating in parallel and sized so that two operating filters can handle the full flow to the circuit. After deaeration, powdered zinc will be added to the PLS solution resulting in the dissolved precious metals precipitating. The zinc rich PLS will be filtered using three recessed plate and frame filters operating in parallel to remove the metal-loaded precipitate. The filtrate or barren solution reports to the barren tank, fortified with sodium cyanide and will be recycled back to the heap. As the precipitate filters accumulate metal-loaded precipitate, they will be taken out of service, cleaned and the precious metal precipitate will be collected in carts. The collected, wet precipitate is then transferred to a mercury retort for drying and removal of any mercury.

After processing in the retort, the dried precipitate will be removed from the retort, weighed and readied for smelting. During the smelting process, the precipitate will be mixed with flux and smelted in an induction furnace. Smelting off gasses are to be collected via a hood and ductwork system and processed via a scrubber system. At the completion of smelting, the furnace contents will be poured where the precious metal doré is separated from the slag generated during the process. The doré will be sampled, weighed and prepared for shipping. The slag is then weighed, sampled and stored for further processing if required, or transferred to the heap.

17.2.2 Phase II: Florida Mountain 2 ktpd Concentrator with Leach Process Description

The second phase of development is the addition of a flotation concentrator with concentrate leach/CCD circuit for the recovery of precious metals in year 3. This circuit is designed to process the Florida Mountain unoxidized ore types. To minimize capital requirements, the CCD PLS will be transferred to the operating heap-leach Merrill-Crowe circuit for recovery of the precious metals. The proposed simplified Florida Mountain 2 ktpd concentrator process flow sheet is shown in Figure 17.2.

Figure 17.2 Florida Mountain: 2 ktpd Flotation with Leach



Florida Mountain ROM material will be crushed to a nominal 100 millimeter size using a jaw crusher for primary crushing. Feed to the jaw crusher will be scalped using a vibratory grizzly with the grizzly reporting to the primary crusher product conveyor. The grizzly oversize reports to the jaw crusher which discharges to the primary crusher product conveyor.

The primary product conveyor reports to the secondary crushing vibratory triple-deck screen operating in closed circuit with the secondary and tertiary cone crushers. Over size from the triple-deck screen reports to the secondary cone crusher. Product from the middle screen reports to the tertiary cone crusher with the screen undersize reporting to the screen product conveyor. The screen product conveyor reports to the mill fine ore bin to supply the milling circuit.

Circulating loads around the secondary and tertiary crushers are modeled at 140% and 233% respectively. The tertiary crusher will operate in a "choke-fed" mode resulting in a final product p80 of 6mm. The crushing plant will operate 14 hours per day for two 8-hour shifts, seven days a week, to supply the mill feed bin at a nominal production rate of 150 tph. The mill feed bin will have enough capacity to feed the downstream milling circuit for eight hours during crusher down days, proposed capacity is 1,000 tonnes.

Mill feed will be reclaimed from the mill feed bin at a nominal production rate of 83 tph using two belt feeders operating in parallel, each being sized to handle 100% of the feed rate. The ore from the belt feeders will report to the mill feed conveyor which in turn will report to the primary grinding mill. The primary grinding mill is planned to be a 4m x 6m wet overflow ball mill. The ball mill will operate in closed circuit with two 250 millimeter hydro-cyclones, with one cyclone operating and one in stand-by to produce a product with a p80 of 212µm.

The grinding circuit cyclone overflow will report to the flotation conditioning tank where the flotation reagents are to be added. Conditioned slurry will report to the first of four rougher flotation cells for a combined retention time of 30 minutes. Rougher concentrate will report to the rougher concentrate pump box, which in turn reports to the cleaner circuit conditioning tank. Rougher flotation tailings will report to the first of three rougher scavenger cells with a combined retention time of 23 minutes.

The rougher scavenger flotation tailings will report to the final tailings pump box and are then transferred to the tailings thickener and thickened to 60% solids by weight. The thickener overflow then reports back to the process water system. The thickened tailings from the thickener underflow are to be pumped to the thickened tailings storage tank, from which they are subsequently pumped to the flotation tailings filter. The tailings filtration filtrate then reports back to the process water system. The filtered tailings are to be transferred to the tailings storage facility and deposited using dry-stacked tailings management procedures.

The concentrate from the first two rougher scavengers will report to the rougher concentrate pump box. The last rougher scavenger concentrate is then recycled back to the head of the rougher scavenger circuit.

The conditioned concentrate slurry will flow to a series of cleaner flotation cells with the combined first cleaner concentrate reporting to the second cleaner flotation cell. Tailings from the first cleaner flotation circuit then report to the rougher conditioning tank. Tailings from the second cleaner flotation cell will report back to the first cleaner conditioning tank. The concentrate from the second cleaner flotation cell then reports to the final concentrate pump box, from which it is the transferred to the flotation concentrate regrind circuit.

The concentrate regrind circuit is to be comprised of a 2 meter x 4.5 meter overflow ball mill that operates in closed circuit with two 100 millimeter hydro-cyclones, one operating and one for standby. The cyclone overflow at a p80 of 50 µm then reports to the pre-leach thickener where the slurry will be mixed with lime and flocculant and thickened to 50% solids by weight. The thickener underflow is to be pumped to the leach circuit comprised of six leach tanks operating in series. The leach tanks are designed to provide a residence time of 4.0 hours each, for a total leach time of 24 hours. The final leached pulp is to be pumped to the CCD circuit to separate the leached solids from the precious-metal rich leach solution. The first stage CCD thickener overflow is to be collected and pumped to the heap-leach Merrill-Crowe circuit and processed to recover the contained precious metals. The final CCD thickener underflow is to be pumped to the heap-leach crushing circuit where the it will be mixed with the fresh heap-leach feed. This mitigates the need for tailings detoxification and separate leached tailings storage facility ("TSF").

17.2.3 Phase III: DeLamar 30 ktpd Heap Leach Process Description

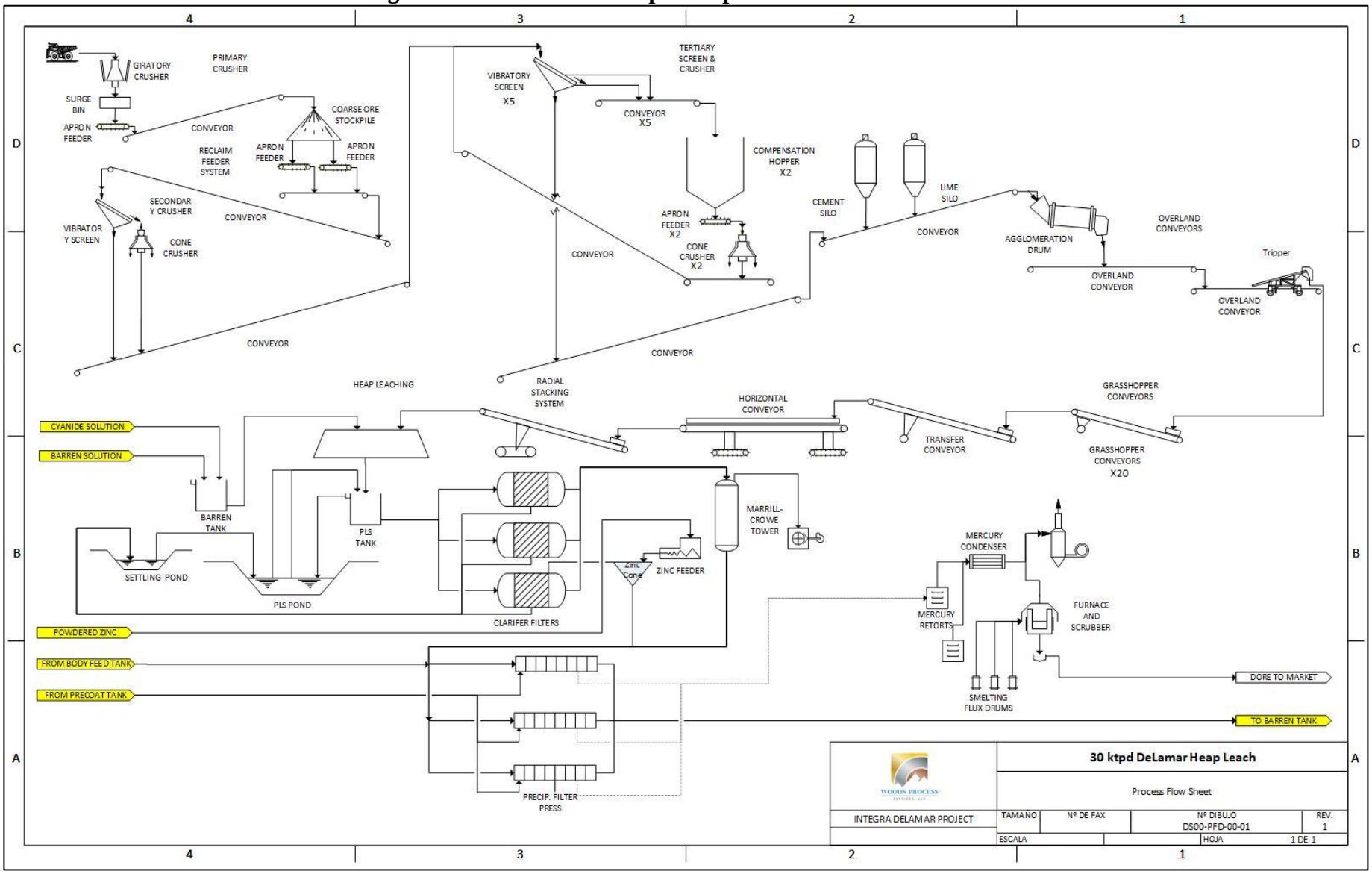
The third phase of process development will be required due to the finer crush size, p80 of 13mm, to process the DeLamar oxide and transition ores by heap leaching. This phase involves the modification of the 30 ktpd crushing and stacking system to include tertiary crush and cement agglomeration (although available data are encouraging and indicate that consideration of two-stage crushing for the DeLamar heap-leach feed will be required). The proposed simplified DeLamar 30 ktpd heap-leach process flow sheet is shown in Figure 17.3. For the sake of brevity, only the modifications to the heap-leach crushing circuit are discussed in this section.

Crushed product from the original heap-leach secondary circuit is to be conveyed to the Phase III tertiary crushing circuit vibratory screen. The oversize product is then conveyed to the tertiary crusher feed bin, with the ore being reclaimed using two belt feeders operating in parallel. Each belt feeder will feed ore to a tertiary cone crusher for a total of two tertiary cone crushers operating in parallel. The undersize from the tertiary crusher screen will report to the final crusher product conveyor where cement for agglomeration is added. The ore/cement mixture then reports to the Phase III agglomeration drum where solution is added to effect agglomeration of the ore. The agglomerated ore will report to the first overland conveyor for transport to the heap-leach pad via the stacking system.

17.3 Process Water, Energy and Materials

Projected requirements for energy and process materials are summarized in Section 21.6. Process water makeup is estimated to be on the order of 75 cubic meters per hour, for a total process requirement of 660,000 cubic meters per year. This does not include infrastructure, facilities or mining use.

Figure 17.3 DeLamar 30 ktpd Heap-Leach Process Flow Sheet



18.0 PROJECT INFRASTRUCTURE

Figure 18.1 shows the DeLamar and Florida Mountain general arrangement drawing. This includes pit designs, waste storage facilities, leach pads, and tailings facilities.

The main access to the project is via gravel roads from Jordon Valley, Oregon, as used for previous mining at DeLamar. While this access is currently deemed to be in good working condition, it will require maintenance during the mine life. Access between the DeLamar mine and Florida Mountain will need to be improved for use with mining equipment. This access will be utilized for delivery of all consumables, as well as any required construction materials and equipment. This will be the primary access for all personnel working at Florida Mountain.

Due to the high altitude and amount of snowfall during winter months, the access roads will be maintained using the mine graders along with a sanding dump truck fitted with a blade for snow removal.

Security fencing will be constructed around the facilities. Buildings will include a site warehouse, administrative offices, an assay lab, and a mine shop.

Electrical power will be supplied by Idaho Power and transmitted to the project via improvements to existing transmission and power lines capable of delivering up to 20MW. The improvements include substation and transmission line upgrades from the Caldwell substation to the DeLamar tap along Highway 234. The powerline from the DeLamar tap to the mine site does not need to be upgraded.

The heap-leach facility will be located between DeLamar and Florida Mountain (Figure 18.1). The crusher is to be located along the haul road between the two deposits. Not shown at this time are conveyors that will transport material from the crushing facility to the leach pad. Facilities for processing of pregnant fluids coming from the leach pad will be located north of the leach pad. Retention ponds will also be located to the west of the pad. The processing facilities and the ponds will be located down-stream from the leach pad to facilitate fluid handling.

18.1 Heap-Leach Pad Construction

The heap-leach pad for Florida Mountain Phase 1 material will be a valley fill located approximately 3.0 kilometers north of the open pit and 600 meters west of the crusher (Figure 18.1). Pad construction will consist of removing growth media, followed by earthwork grading to achieve uniform contours to apply lining and solution collection systems. The lining system will generally consist of a composite liner with high density polyethylene ("HDPE") placed over a compacted clay under-liner. Perforated pipes will be located immediately above the HDPE liner to collect the metal laden leach solutions and convey them to the downstream toe of the heap where they will be collected in a head tank for pumping to the processing facilities. Leak detection will be installed beneath the lining system to collect any seepage in the event that any leaks should occur in the composite lining system. All solutions containing cyanide will be transported in double-lined pipe to minimize the possibility of spills from pipe rupture.

The heap-leach pad is scheduled to be constructed in two phases – the first phase during preproduction and the second phase in year 3. As described in Section 17.0, the ore will be crushed and stacked on the

heap with conveyors. The overland and portable conveyors will be installed in Phase 1. Phase 2 construction doubles the area of the combined lining system. The Phase 2 expansion will be sufficient to provide capacity for the estimated heap-leach resources from both Florida Mountain and DeLamar mining areas.

18.2 Dry Stack Tailings Construction

Tailings disposal will involve stacking dewatered (filtered) tailings into a valley fill site approximately 500 meters north of the processing facilities (Figure 18.1). Preparation of the site will be similar to the heap-leach pad construction with grading, placement of a compacted clay under-liner, placement of an HDPE primary liner, and a leak detection under the composite liner system. The tailings will be transported to the dry stack site with a conveyor and spread with a small dozer into thin lifts to promote additional drying and consolidation. The fill will be constructed from the bottom upward with the face of the fill covered annually with growth media and revegetated for erosion protection and reclamation. Initial placement in the valley bottom may be augmented with small haul trucks or scrapers to reduce frequency of moving portable conveyors. Meteoric water on the fill surface will be collected and pumped to the process area for makeup water, or will be treated and released from the existing treatment plant depending on makeup water demand.

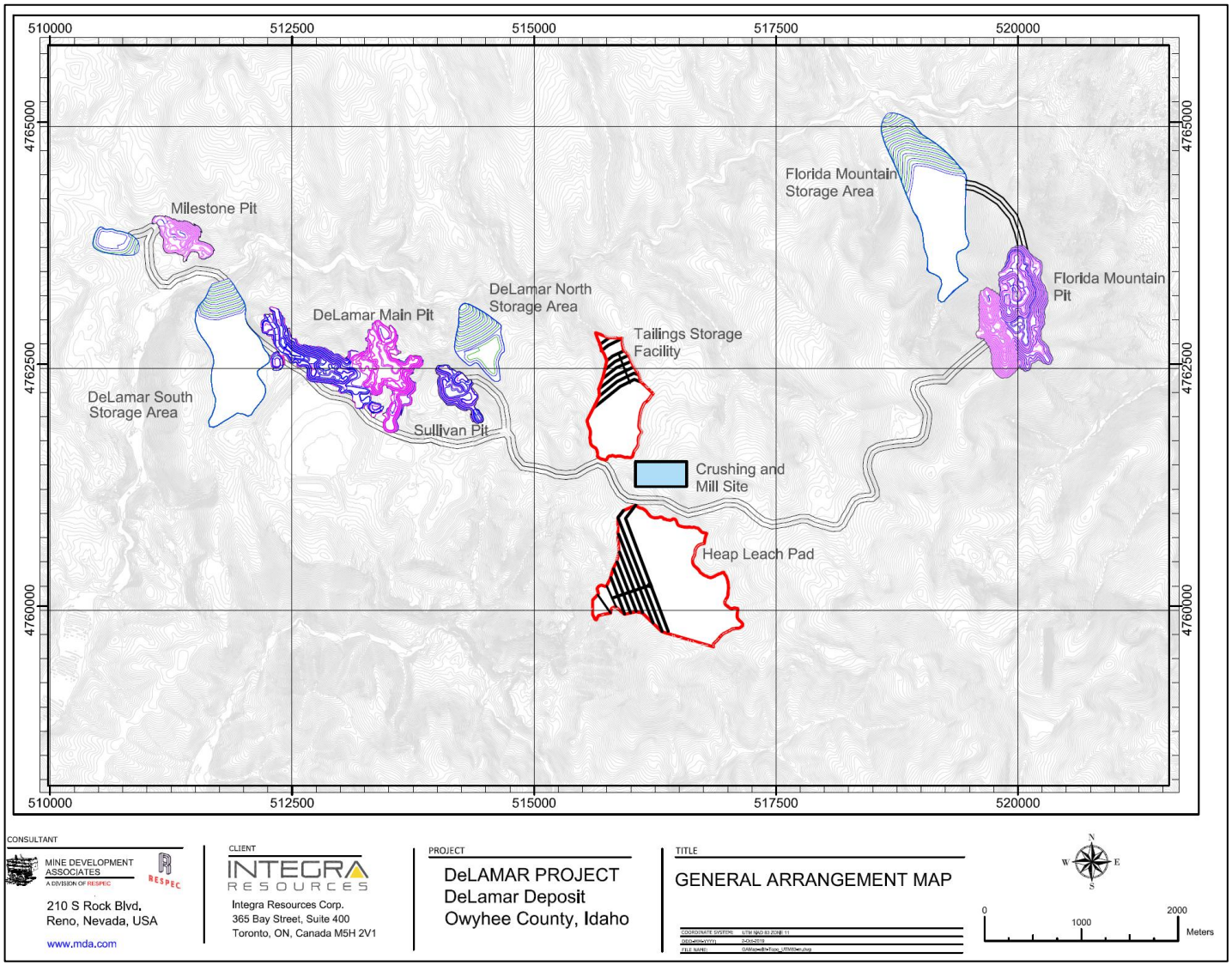
18.3 Mine Site Personnel

Mine site personnel requirements are shown in Table 18.1. This includes administrative, mining, and processing. In addition, there would be approximately 80 additional personnel working on site during construction.

Table 18.1 Mine, Process and Administrative Personnel

	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Max
Administration	#	24	27	24	24	24	24	24	24	24	24	24	-	27
Mining General Personnel	#	22	22	22	22	22	22	22	22	22	22	22	-	22
Mine Operations Hourly Personnel														
Operators	#	40	79	87	107	107	111	85	71	71	71	67	-	111
Mechanics	#	20	41	43	55	55	59	43	37	37	37	33		59
Maintenance	#	25	25	25	25	25	25	25	25	25	25	25	-	25
Mineral Processing - Heap Leach	#		74	74	74	74	74	74	74	74	74	74	-	74
Mineral Processing - Milling	#				41	41	41	41	41	41	41		-	41
Total	#	131	268	275	348	348	356	314	294	294	294	245	-	356

Figure 18.1 PEA General Arrangement Drawing



19.0 MARKET STUDIES AND CONTRACTS

19.1 Metal Pricing

No market studies have been undertaken for this PEA. Gold doré will be the commercial product from the DeLamar and Florida Mountain operation. Gold doré is readily sold on the global market to commercial smelters and refineries, and it is reasonable to assume that doré from the project will also be salable.

To determine appropriate metal prices to be used for economic analysis and cutoff grades, Mr. Dyer has considered spot prices in the months prior to the effective date of this report and reviewed current metal prices used in recent NI 43-101 technical reports. In addition, three-year trailing gold and silver prices were reviewed along with one-year forward pricing.

Economic analysis is discussed in Section 22 and uses a gold price of $1,350 per ounce and a silver price of $16.90.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Baseline Studies

Integra has contracted qualified third party(ies) to perform environmental adequacy reviews of all available existing environmental baseline reports and data compiled from 1979 through present. Additionally, two environmental impact statements ("EIS"s) completed prior to 1982 by previous operators for the site were approved.

In 2019, Integra intends to conduct technical adequacy audits of all existing environmental information, and to develop individual work plans for supplemental studies to include the following:

- Surface water hydrology and quality;
- Ground water hydrology and quality;
- Geochemistry;
- Water rights;
- Soils and vegetation;
- Geotechnical/engineering;
- Wetlands and riparian resources;
- Fisheries and aquatics;
- Wildlife;
- Cultural and Tribal resources;
- Inventoried roadless area(s);
- Transportation and public safety;
- Recreation and visual;
- Air quality/noise;
- Hazardous materials;
- Reclamation;
- Previous mining operations; and
- Socio-economics.

Eventually, a BLM interdisciplinary team will review and approve the environmental baseline work plans for the resources listed above. The BLM interdisciplinary team is specifically organized to oversee these environmental studies, which would be a key element of an EIS. It is comprised of highly qualified specialists in each of the resource categories. Initial supplemental baseline studies for surface and

ground water, wetlands, and vegetation will be initiated in the summer of 2020. Geotechnical and geochemical fieldwork were also commenced during the 2019 season.

The environmental baseline program for all major resource categories would continue through 2022 to allow a "full and fair" discussion of all potentially significant environmental impacts of an EIS. The entire DeLamar mining district has been studied extensively, both historically and currently; therefore, ensuring scientific integrity of the methodologies and analysis used to collect the data and ultimately a meaningful analysis would be conducted allowing for a reasonable comparative assessment of the alternatives.

20.2 Permitting

20.2.1 Environmental Impact Statement

Approval of any Final Plan of Operations/Reclamation Plan for the project requires an environmental analysis under the National Environmental Policy Act ("NEPA"). NEPA requires federal agencies study and consider the likely environmental impacts of the proposed action before taking whatever federal action is necessary for the project to proceed.

The purpose and need for the project would be to conduct open pit mining, which would disturb over 1,000 acres of land on unpatented and patented mining claims within the project area, to produce silver and gold from mineralized material of the estimated mineral resources.

The EIS serves as an "overarching" federal permit requirement, as well as that of at least three other likely federal authorizations:

- IPDES Permit for water discharge;
- USACE 404 Dredge and Fill Permit; and
- ESA Biological Opinion.

The EIS Record of Decision ("ROD") effectively drives the entire permitting process timeline. These important clearances cannot be obtained without a favorable ROD.

The Council on Environmental Quality identifies ten factors for determining the significance of a proposed action. The three primary circumstances for mining activities being:

- Potential impacts on wilderness and other pristine, undeveloped areas;
- Potential impacts on threatened and endangered species; and
- Situations where several individual mining projects would affect a single watershed circumstance.

The bullets above, along with the fact that the project site is a "brownfield site" (extensively mined in the past), combine to make any proposed Integra action of developing a mine and processing facilities at the site subject to an EIS. Other primary federal and state authorizations and/or permits are described in

the following sections, which ties the EIS and other permits together in terms of an estimated schedule and costs for completing the program.

20.2.2 Idaho Pollutant Discharge Elimination System Permit ("IPDES")

An IPDES Permit would be required for point source discharges from the mining operation to "waters of the United States." Likely point discharges would include treated mine drainage, treated net precipitation from the tailings storage facility, and any other discernible or discrete point source associated with mining and processing at the site. In addition, the project would be subject to performance standards for new sources for its respective industrial source category. This means the project would have to demonstrate that it is applying the best available control technology to meet applicable water quality standards. The permit application must be submitted at least 180 days prior to the approved discharge.

Storm water discharges associated with this industrial activity require a related permit. Storm water is defined as "storm water runoff, snowmelt runoff, and surface runoff and drainage." Where flows are from conveyances that are not contaminated by contact with overburden or other mine waste, a permit is not required. Hence, the water management scheme developed for the project endeavors to collect and convey clean water around the mining operation and discharge downstream. Active storm water would be managed via a storm-water pollution prevention plan, which must also be submitted at least 180 days before commencing the discharge.

20.2.3 U.S. Army Corps of Engineers Section 404 Dredge and Fill Permit

A Section 404 Permit is required under the Clean Water Act for the discharge of dredged or fill material into waters of the United States. Dredged or fill material includes tailings and waste rock. Other activities, in addition to the tailings and waste rock storage that may require a 404 Permit are:

- Road construction;
- Bridges;
- Construction of dams for water storage;
- Stream diversions; and
- Certain reclamation activities.

Waters of the United States include wetlands. A 2010 U.S. Supreme Court decision found mine tailings to be "fill," and can, therefore, be placed in waters of the United States with an approved Section 404 of the USACE Dredge and Fill Permit.

20.2.4 Biological Opinion

The ESA prohibits the taking of fish and wildlife species classified as endangered or threatened, unless otherwise authorized. The following species may be in the vicinity of the DeLamar site:

- Redband trout;
- Bull trout;

- Whitebark pine (sensitive); and
- Milkvetch (sensitive).

A biological opinion would likely be required for the listed species, under Section 7 of the Clean Water Act. Federal agencies are required to "conserve endangered or threatened species, and to ensure that their actions are not likely to jeopardize the continued existence of any of these species or adversely modify their designated habitat" (ESA, 16 U.S.C. Section 1538(a)). If present, consultation would be with the U.S. Fish and Wildlife Service (USFWS).

There is no comprehensive federal ground water quality statute, in contrast to surface water and the Clean Water Act. Ground water protection is found in several programs, which include the Safe Water Drinking Act, sections of CERCLA, and the Resource Conservation and Recovery Act. The project may need an Underground Injection Control Permit, if it elects to land apply treated mine-waste water, which involves the subsurface emplacement of fluid by well injection. The Safe Water Drinking Act was implemented by the State of Idaho to enforce drinking water regulations for the related facilities. Based on the anticipated number of personnel, this operation would be classified as a public water system.

The federal Clean Air Act regulates air quality and the project would be subject to National Ambient Air Quality Standards; definitive air quality criteria would apply. The project would be required to meet Prevention of Significant Deterioration requirements, visibility regulations, and National Emission Standards for Hazardous Air Pollutants. This would involve pre-construction and operating permits described in the following section.

20.2.5 Major State Authorizations, Licenses, and Permits

The key authorizations, licenses, and permits required by the State of Idaho are summarized in this section. The federal and state application processes would be integrated and processed concurrent with the EIS.

- Air Quality Application for Permit to Construct and Operate – Assesses the allowable impacts to air quality and prescribes measures and controls to reduce and/or mitigate impacts.
- Cyanidation Permit – Required by the IDEQ and is applicable for a facility that processes mineralized material using cyanide as the primary reagent. Integra contemplates processing the gold and silver mineralized material at a heap leach facility(s) and a mill with associated tailings facility. The regulations apply to both operations and closure and reclamation of any facility.
- Land Application Permit – In order to apply any treated process wastewater to a designated land area for ultimate disposal, the mining company must obtain a Land Application Permit from IDEQ. The project would need to meet the performance standards for new sources “zero discharge” requirement for net precipitation minus evaporation to ensure no unpermitted discharges.
- Ground Water Rule – This rule establishes minimum requirements for ground water protection through standards and a set of aquifer protection categories. To implement the rule, Integra would need to request to establish points of compliance outside and

down-gradient from the mine area(s). Integra would also establish reasonable upper-tolerance limits for all compliance wells, working directly with IDEQ. These upper-tolerance limits would take into account the high naturally occurring background levels for several parameters.

- Total Maximum Daily Loads (TMDL) – Jordan Creek water quality is considered "impaired." IDEQ has established TMDL that set maximum pollutant levels for temperature.
- Water Rights – There are eight permanent water rights associated with the mining activity area. In addition to these established rights, Integra holds three temporary rights covering both surface and ground water that can be renewed annually.
- Stream Channel Alteration Permit – Required by the IDWR for a modification, alteration, or relocation of any stream channel within or below the mean high water mark. The PEA contemplates relocating one or more unnamed creeks, both temporarily and permanently, as part of the overall mine plan.
- Dam Safety Permit – The IDWR requires a Dam Safety Permit for dams greater than ten feet high or for reservoirs exceeding a 50-acre-feet storage capacity. The Application to Construct a Dam includes design plans and specifications for construction of the dam. Mine tailings impoundments greater than or equal to 9.14 meters (30 feet) high are regulated by IDWR in the same manner. Design and Construction Requirements for Mine Tailings Impoundment Structures are described in IDAPA 37.03.05. The PEA contemplates construction of a TSF in an unnamed creek drainage and would need authorization.
- Water and Wastewater Systems – The drinking water system(s) design for the contemplated facilities must be approved prior to use to ensure compliance with the Safe Drinking Water Act. IDEQ would also require approval of plans and specifications for any new sewage treatment and disposal for the work camp.
- Fuel Storage Facilities – Any proposed fuel storage must also comply with IDEQ design and operating standards, as well as Idaho State Fire Marshall and Owyhee County requirements.
- Reclamation Plan – All surface mines must submit and obtain approval of a comprehensive reclamation plan (Title 47) for mining activities on patented land. The Reclamation Plan includes detailed operating plans showing pits, mineral stockpiles, overburdened piles, tailings ponds, haul roads, and all related facilities. The Reclamation Plan must also address appropriate BMPs and provide for financial assurance in the amount necessary to reclaim those mining activities. The plan must be approved prior to any surface disturbance. A large portion of the planned Florida Mountain and DeLamar pits are located on patented land.
- State Historic Preservation Office – The project is located within the DeLamar National Historic District; therefore, approval of a historic/cultural resources assessment by the State Historic Preservation Office would be required.

- Others – State requirements would also involve compliance with the Idaho Solid Waste Management Regulations and Standards, transportation safety requirements enforced by the Idaho Public Utilities Commission, and others.

20.2.6 Local County Requirements

There are several other permits and approvals that would apply to the project, if it proceeded to a full-scale mining proposal, including:

- Conformance with the Owyhee County Comprehensive Plan;
- Issuance of building permits by the county; and
- Sewer and water systems approval by Central District Health Department, and various other authorizations.

Additionally, an annual authorization by the Owyhee County Road Department for an Owyhee County Road Use Permit for any mining operation is essential. The permit addresses standard operating procedures for the road route to be used, seasonal limits, spill prevention and response planning, HAZWOPER or hazardous materials handling training, convoying, and other requirements.

Integra has not entered into negotiations or agreements with local communities.

20.2.7 Idaho Joint Review Process

The IDL is responsible for implementation of the Idaho Joint Review Process, which was established in order to coordinate and facilitate the overall mine permitting process in the state. It involves an interagency Memorandum of Understanding ("MOU") between involved state and federal agencies and addresses a process to achieve pre-analysis coordination in approving/ administering exploration permits, interagency agreement on plan completeness, alternatives considered, draft and final permits, bonding during mine plan analysis, and interagency coordination related to compliance, permit changes and reclamation/closure for major mining projects. In Idaho, the Joint Review Process was established as the basis for interagency agreement (state, federal, and local) on all permit review requirements. The focus of the process is concurrent analysis timelines. This would include, for example, in the case of the DeLamar project the NEPA process, IPDES permit, USACE 404 permit, state 401 Certification of these two key permits, the state Cyanidation permit, and the ESA Consultation. The Idaho Joint Review Process would play a key role in achieving two primary permitting goals: 1) increased communication and cooperation between the various involved governmental agencies, and 2) reduced conflict, delay, and costs in the permitting process.

20.2.8 EIS / Permitting Timelines and Costs

In order to develop estimated schedules and costs for the anticipated EIS requirement, recent EIS and permitting for a large open pit mine at Rochester, Nevada, and the Kennecott Utah Copper Tailings Expansion EIS in Utah were reviewed. Rochester was successfully permitted, and Kennecott is currently undergoing the EIS.

Rochester was evaluated for the scale of the operation. Collectively, these projects presented the full array of environmental permitting, including EIS's and/or assessments, air quality, processing using

cyanide, EPA IPDES water discharge permits, water rights, USACE 404 permitting, diesel fuel transport and contingency plans, tailings dams, surface and ground water protection, and power requirements. Other issues involve fish passage permits, storm water, native consultation, cultural resources, and others. Evaluations of all of these items would be required in the EIS for the project.

All of these projects provide a good look back perspective. However, what they do not consider is the Department of Interior's (DOI's) Secretarial Order 3355, which requires a one-year timeline for EISs completed by the DOI agencies, such as the BLM. The one-year timeframe commences with the publication of the Notice of Intent in the Federal Register and ends with the signing of the ROD. The discussion below assumes that the BLM will be the lead federal agency for NEPA, and that the USACE will be a cooperating agency. With regard to the likely scope of DeLamar, the following conceptual description was developed as the basis for this permitting analysis:

- Regulatory – EIS required; BLM Lead Agency; EPA, USACE, and IDL are cooperating agencies;
- Mining– estimated at 30,000 t/d mineralized material with a 1.1:1 waste to mineralized material strip ratio;
- Processing – tailings by-product (flotation and comingled concentrate tailings) with high energy requirement for oxidation of the mineralized material; heap leaching is also anticipated for the project;
- Power – line power would be utilized for initial development and expanded to meet further power needs;
- Waste Rock – some selective placement would be required likely due to anticipated geochemical reactivity; large volumes would be stored and managed;
- Water Supply – available from existing water rights;
- Water Treatment – high technology with a high energy requirement (likely reverse osmosis or similar);
- Project Access – existing Jordan Creek road from Jordan Valley, Oregon;
- Manpower – up to 440 direct and indirect jobs during construction; estimated 360 during operations;
- Operating Schedule – mining and processing year-round; and
- Total Land Disturbance – over 1,000 acres of patented and unpatented land (potential land exchange).

This concept was developed only for the purpose of scaling the project, such that the estimated schedules and costs could be compared with the projects listed earlier.

An EIS/permitting timeline is summarized below in five primary permitting windows.

- Initial 12 to 24 Months – Start baseline confirmatory studies for surface and ground water, geochemistry, fisheries and wildlife, geotechnical, as well as air quality and

wetlands work. Negotiate an MOU with the BLM for preparing the EIS. Conduct initial internal scoping with "high up" agency and political contacts.

- Months 20 to 30 – Commence preparation of the Initial Plan of Operations. Concurrently, develop all other permit applications for submittal. During this period, the third-party EIS Contractor would be selected by the BLM and Integra with input from the USACE and EPA. This would occur in time for the contractor to lead the scoping meetings. The contractor would then finalize the EIS work plan and initiate early environmental baseline adequacy determination write-ups for the various resource categories (air, water, socio-economics, etc.).
    - The contractor would also write the alternatives section of the EIS. This is a significant section that must present only reasonable and potentially feasible alternatives. Integra would file its Initial Plan of Operations with the BLM mid-way through this window at around Months 25 to 27, which would trigger the EIS.
    - Integra would have completed a preliminary feasibility study during this timeline or permitting window. Once this is done, Integra can narrow down the best alternative both from a cost and environmental standpoint. This level of pre-feasibility information would also be crucial to success in obtaining the IPDES water discharge and USACE 404 permits. The USACE 404 Permit would be needed for wetlands disturbance, and any stream diversions needed for the project.
- Months 30 to 34 – A preliminary draft EIS would be completed by the BLM (Third-Party Contractor). This document would be for the lead and cooperating agencies and Integra review only. Typically, this review would require about 30 to 60 days. In the initial stages of this period, Integra would file most, if not all, of their permit applications. Some, like the water rights applications, if needed, would have already been submitted to the appropriate agencies. Others, like the USACE 404 Permit and EPA IPDES Permit, cannot be issued until after the final EIS and ROD by the BLM have been issued.
- Months 34 to 42 – A draft EIS would be produced for public review. The review period would be about 60 days.
- At Months 40 to 42, the final EIS would be issued. At this point, the BLM could choose to issue the ROD concurrently or elect to issue it 30 days later. There would be an administrative appeal period involved at this point. For the purposes of this very preliminary assessment, an additional 60 days could be used, pushing the project out to Month 44. The remaining permits would also be issued over this period.
- Summary Best Case Schedule – Estimated at 40 to 42 months (3.5 years) with a concurrent baseline data collection program.

20.2.9 Most Likely Case EIS Cost Summary

The costs listed below are factored from real costs. This is considered the most realistic estimate given the various permitting uncertainties associated.

- EIS "Project" – $1.4M (represents all costs for third-party EIS and BLM reimbursement);
- Support Engineering – $750,000 (does not include pre-feasibility study);

- Legal – $500,000;
- Potential Baseline Study Needs –$2.8M to $4.0M, depending on ground water and geochemistry issues;
- Permitting "Project" – $2.5M (represents all costs for separate permitting program); and
- Total Estimate – $8.0M to $10.2M.

20.2.10 Integra Permitting Management Strategy

To successfully achieve any such permitting program, estimated costs, and timeline, Integra has designed a seven-point management process that includes the following key points:

- MOU providing for agency cooperation, accountability, and predictability;
- Requirements for quality consultants;
- Communication plan for the consultants;
- Baseline studies, adequacy determinations and tracking procedures, EIS completeness;
- Budget and schedule tracking and cost controls;
- Goals for environmental enhancement in mine planning and closure; and
- An informed public affairs process.

20.2.10.1 Permitting Risks and Risk Management Strategy

This section summarizes certain environmental issues and risks, and strategies by Integra to manage and/or mitigate these risks. The overall approach is a two-part strategy that involves a proactive regulatory/governmental affairs program, which has already been initiated by the company, and a supplemental environmental baseline program that clearly measures pre-existing conditions at the site. The description which follows highlights those risks. It also lists the measures Integra has put in place to avoid permitting delays and adverse outcomes.

There is a risk that any single environmental issue or combination thereof could delay the permitting process. To date, Integra has incorporated specific standard operating procedures and best management practices into their exploration plans. These programs would be carried over by Integra to any full-scale mining operation.

There is a risk that the IPDES permit for water discharges from the operation would impose stringent water quality criteria. Integra plans a three-tier system of BMPs, standard operating procedures, and state-of-the-art water treatment to meet these criteria. The water treatment facilities contemplated in this PEA have been proven at other mining operations located in very sensitive environments.

Integra's risk management strategy focuses on a three-pronged approach. First, their development program highlights adequacy of the environmental baseline, as discussed earlier. Second, establish an open dialogue with key environmental organizations, tribal governments, and involved agencies. This would include meetings, site-visits, and project previews with these groups. Third, would implement a

"litigation avoidance initiative" to be formulated with the key environmental and tribal organizations. This could involve: 1) some level of joint-operational monitoring; 2) input to reclamation planning; 3) employment and business opportunities for the tribes; 4) third-party environmental audits; and 5) certain other considerations. The objective is to make the project a fully integrated, sustainable, and socially and environmentally responsible operation through open communications and accessibility.

The overall permitting process could be expedited. This would involve negotiating specific permitting agreements or MOU's with involved agencies. These MOU's would serve as an organizational framework for establishing agency and project proponent working agreements. The agreements would cover EIS and/or major permits like the EPA IPDES Permit for water discharge and the USACE 404 Dredge and Fill Permit. These two permits are required for water discharge from the tailings facility, and any discharge from waste rock storage facilities. The MOUs would address: 1) organizational contacts and communication procedures/ limitations; 2) NEPA or permitting objectives; 3) work required to achieve EIS or permit completion; 3) third-party involvement; 4) statement of responsibilities; 5) deliverables; 6) importantly schedules for review and completion; 7) coordination needs including consultation (Section 106 Historical Consultation and Native Consultation); 8) public involvement requirements; and 9) legal requirements.

20.3 Social and Community

The project is located in rural Owyhee County, close to the Oregon border. The closest substantial community is Jordan Valley, in Malheur County Oregon. This community is primarily an agricultural-based economy. However, when the mine previously operated in the 1980s and 1990s many of the employees lived in Jordan Valley.

20.4 Waste Characterization

Integra's consultant will be conducting a mine waste characterization program as part of the planning and impact assessment for the project. Geochemical testing of mine waste materials provides a basis for assessment of the potential for ML/ARD, prediction of contact water quality, and evaluation of options for design, construction, and closure of the mine facilities. This work also supports the next phase of the project's potential advancement, including environmental assessment and permitting. The characterization effort focuses on the assessment of waste rock geochemistry, evaluation of tailings material from mineral benefaction processes, and determination of final pit wall geochemistry.

Geochemical characterization is an iterative process and sample collection for the project is being completed in phases. The first phase is complete and involved the collection of samples from core generated during the recent exploration drilling activities. Subsequent phases of the characterization program would focus on improving the spatial representation of the dataset as drill core from the ongoing exploration and geotechnical drilling becomes available.

20.5 Closure and Reclamation Strategy

A comprehensive reclamation and closure plan would be developed for all disturbances and infrastructure associated with the project. Reclamation objective standards established by industry best practices and regulatory requirements for reclamation would be fulfilled. Integra fully understands the

importance of the trout fisheries and temperature issues that exist in the Jordan Creek drainage, as well as related environmental closure needs required to sustain and/or improve these resources as part of any overall mine and closure plan. Doing so would involve a "mine for closure strategy" that begins with the end in mind (i.e. Integra would seek to develop an economic mine plan and closure/reclamation strategy that integrates certain fisheries' habitats and restoration components). The plan would meet all standards of the Clean Water Act. At the end of mining, the plan would result in a sustainable improvement in the fishery and other aquatic and riparian habitats. The plan would also mitigate wetland impacts to recreate, enhance or replace productive wetlands and other wildlife habitats.

The goals of this reclamation and closure plan are expected to evolve based on cooperative discussions, public and regulatory input; however, the initial goals include:

- Protecting water quality and the fishery;
- Restricting or eliminating the migration of potential contaminants of concern from all sources on the contemplated mine site;
- Restricting or eliminating potential public safety risks associated with the potential decommissioned and reclaimed mine site;
- Restoring the property, to the extent possible, to pre-mining conditions or better; and
- Improving the property by incorporating environmental enhancement projects, which may include wetlands and riparian creation, fish and upland habitat enhancement, replanting of current and future disturbed areas to enhance wildlife habitat and reduce sedimentation, enhanced reclamation of historic disturbance that is currently contributing sedimentation and/or ML into the environment, etc.

21.0 CAPITAL AND OPERATING COSTS

Capital and operating costs were estimated for the PEA by MDA (mining and infrastructure), Mr. Jeffrey Woods, principle consulting metallurgist at Woods Process Services (Processing), and Mr. John Welsh, senior principal of Welsh Hagan Associates (leach pads and tailings impoundment). Table 21.1 summarizes the estimated capital costs for the project. The LOM total capital costs are estimated as $270.3 million, including $161.0 million in preproduction and $109.3 million for sustaining capital. Sustaining capital includes $20.0 million in reclamation costs. Capital costs below are inclusive of sales tax, engineering, procurement and construction management ("EPCM") and contingency.

Table 21.1 Capital Cost Summary

Mine	Pre-Production (1)	Sustaining Yr 1 to Yr 10 (1)	Total LOM (1)
Mining Equipment	$ 32,980	$ 52,014	$ 84,994
Pre-Stripping	$ 7,514	$ -	$ 7,514
Other Mine Capital	$ 6,027	$ 746	$ 6,773
Sub-Total Mine	**$ 46,521**	**$ 52,760**	**$ 99,281**
Processing			
Heap Leach Pad	$ 14,130	$ 19,178	$ 33,308
Heap leach Plant (Incl Crushing and Stacking)	$ 48,449	$ -	$ 48,449
Heap leach: Agglomeration / Crushing (DeLamar Ore)	$ -	$ 20,518	$ 20,518
Florida Mill: Plant	$ -	$ 34,354	$ 34,354
Florida Mill: Dry Stack Tailings	$ -	$ 6,990	$ 6,990
Sub-Total Processing	**$ 62,579**	**$ 81,039**	**$ 143,618**
Infrastructure			
Power	$ 21,714	$ -	$ 21,714
Assay Lab	$ 2,804	$ -	$ 2,804
Other	$ 2,552	$ 974	$ 3,526
Sub-Total Infrastructure	**$ 27,070**	**$ 974**	**$ 28,044**
Owner's Costs	$ 5,819	$ -	$ 5,819
SUB-TOTAL	**$ 141,989**	**$ 134,773**	**$ 276,761**
Other			
Working Capital (2)	$ 13,024	$ (13,024)	$ -
Cash Deposit for Reclamation Bonding (3)	$ 6,000	$ (6,000)	$ -
Salvage Value (4)	$ -	$ (26,426)	$ (26,426)
TOTAL	**$ 161,013**	**$ 89,323**	**$ 250,336**
Reclamation	$ -	$ 20,000	$ 20,000
Total Including Reclamation Costs	**$ 161,013**	**$ 109,323**	**$ 270,336**

Notes:

(1) Capital costs include contingency and EPCM costs;
(2) Working capital is returned in year 11;
(3) Cash deposit = 30% of bonding requirement. Released once reclamation is completed;
(4) Salvage value for mining equipment and plant.

Table 21.2 shows the estimated LOM operating costs for the project. Operating costs are estimated to be $7.82 per tonne processed for the LOM. This includes mining costs which are estimated to be $2.00 per tonne mined. The total cash cost is estimated to be $619 per ounce of gold equivalent and all-in sustaining costs are estimated to be $742 per ounce of gold equivalent.

Table 21.2 Operating and Total Cost Summary

	USD / Tonne	
LOM Operating Costs	**Mined**	**Processed**
Mining	$ 2.00	$ 4.18
Processing		$ 3.08
G&A		$ 0.55
Total Site Costs		**$ 7.82**

LOM Cash Costs and All-in Sustaining Costs	By-Product (1)	Co-Product (2)
Mining	$ 380	$ 317
Processing	$ 280	$ 233
G&A	$ 50	$ 42
Total Site Costs	$ 711	$ 592
Transport & Refining	$ 13	$ 11
Royalties	$ 17	$ 14
Total Cash Costs	**$ 741**	**$ 617**
Silver By-Product Credits	$ (272)	$ -
Total Cash Costs Net of Silver by-Product	**$ 469**	**$ 617**
Sustaining Capital	$ 131	$ 109
Reclamation	$ 19	$ 16
All-in Sustaining Costs	**$ 619**	**$ 742**

Notes:
(1) By-Product costs are shown as US dollars per gold ounces sold with silver as a credit; and
(2) Co-Product costs are shown as US dollars per gold equivalent ounce.

21.1 Mining Capital

Mining capital estimates assume owner operations of mining equipment and were based on the equipment and facilities required to achieve the production schedule. Capital costs are based on estimation guides and recent costs for similar projects. The mining capital estimate is summarized by year in Table 21.3.

Table 21.3 Mining Capital Cost by Year

Total Mine Capital	Units	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Total
Primary Mining Equipment	K USD	$ 23,119	$ 24,274	$ 2,968	$ 13,621	$ -	$ 7,494	$ -	$ -	$ 71,476
Support Equipment	K USD	$ 8,113	$ 3,498	$ -	$ -	$ -	$ -	$ -	$ -	$ 11,611
Blasting Equipment	K USD	$ 405	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 405
Mine Maintenance Eqipment	K USD	$ 1,342	$ 159	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,501
Other Mine Capital	K USD	$ 6,027	$ 19	$ 2	$ 8	$ -	$ 317	$ 400	$ -	$ 6,773
Total Mine Equipment Capital	K USD	$ 39,007	$ 27,950	$ 2,970	$ 13,629	$ -	$ 7,811	$ 400	$ -	$ 91,766
Prestripping Costs	K USD	$ 7,514	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 7,514
Total Mine Capital	K USD	$ 46,521	$ 27,950	$ 2,970	$ 13,629	$ -	$ 7,811	$ 400	$ -	$ 99,281

21.1.1 Primary Equipment

Primary equipment purchases refer to the purchase of drills, loading equipment, and haul trucks. The total LOM primary equipment cost estimate is $71.5 million which includes:

- $1.4 million for pioneering drills;
- $7.0 million for production drills;
- $5.0 million for a large loader;
- $13.6 million for hydraulic shovels; and
- $44.5 million for 136-tonne capacity haul trucks.

21.1.2 Support Equipment

Support equipment includes the equipment required to support the primary mining equipment. This includes dozers to manage dumping locations and cleanup of benches for drilling and loading equipment. This also includes road maintenance equipment such as water trucks and graders. The total estimated capital for support equipment is $11.6 million and includes:

- $4.7 million for dozers of various sizes;
- $2.7 million for motor graders;
- $3.6 million for water trucks;
- $30,000 for in-pit pumps to control runoff water;
- $500,000 for a 50-ton capacity crane (to be shared between mining and process); and
- $57,000 for a flatbed truck used for moving maintenance items within the mine.

21.1.3 Blasting Equipment

Blasting equipment includes explosives trucks for use in loading blast holes and a skid loader to be used for stemming holes. The cost estimate for blasting equipment is $405,000 which includes $278,000 for one explosives truck and $127,000 for a skid loader.

21.1.4 Mine Maintenance Capital

The cost estimate for mine maintenance capital is $1.5 million, which includes one large lubrication/fuel truck at $1,007,000; two mechanic's trucks totaling $358,000, and one tire truck totaling $176,000.

21.1.5 Other Capital

Other capital includes an assortment of equipment and facilities totaling $6.8 million. This includes:

- $80,000 for light plants;
- $200,000 Preparation for explosives storage site;
- $79,000 for ANFO storage bins;
- $12,000 for powder magazines to store boosters;
- $7,000 for a cap magazine;
- $62,000 for mobile radios in equipment and assorted handheld radios;
- $750,000 for general shop equipment including hoists and other tooling;
- $105,000 for engineering computers, plotters, and other office equipment;
- $20,000 for dust suppression storage bladders;
- $150,000 for surveying equipment and GPS base stations;
- $53,000 for fuel island facilities;
- $3.5 million for shop and office facilities;
- $800,000 in access roads to each deposit and site preparation;
- $150,000 for ambulance and firefighting equipment; and
- $806,000 for light vehicles.

The access roads and shop are described in Section 18.0.

21.1.6 Mine Preproduction Costs

Mine preproduction is considered as the cost of all mining prior to the start of gold production from the leach pad. For the PEA this is a 6-month period. The total mining costs during preproduction total $7.5 million.

21.2 Process Capital

Process plant capital costs are based on equipment sizing using the design criteria and mass balance, and using the Infomine® mine and mill cost estimators guide. Installed capital costs for civil work, concrete, structural steel, piping, controls and instrument. were factored based on equipment costs. Power distribution system costs were estimated using the installed horsepower estimates.

Leach pad capital cost is based on preliminary engineering, material take offs and associated local unit rates.

Yearly capital costs for heap leach processing are summarized in Table 21.4 based on costs provided by Mr. Woods and Mr. Welsh. Table 21.5 shows the yearly capital costs for the Florida Mountain concentrator also based on costs provided by Mr. Woods and Mr. Welsh.

Table 21.4 Yearly Heap Leach Processing Capital Costs

Leach Pad Construction Cost	**Units**	**Pre-Prod**	**Yr_1**	**Yr_2**	**Yr_3**	**Yr_4**	**Yr_5**	**Yr_6**	**Yr_7**	**Yr_8**	**Yr_9**	**Total**
Pad Construction	K USD	$ 10,304	$ -	$ -	$ 9,000	$ 342	$ -	$ -	$ 6,000	$ -	$ -	$ 25,646
Recovery Plant & Leach Systems	K USD	$ 1,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,000
Total Leach Pad Directs	K USD	$ 11,304	$ -	$ -	$ 9,000	$ 342	$ -	$ -	$ 6,000	$ -	$ -	$ 26,646
Pad Construction Contingency	K USD	$ 2,261	$ -	$ -	$ 1,800	$ 68	$ -	$ -	$ 1,200	$ -	$ -	$ 5,329
Pad Construction ECPM	K USD	$ 565	$ -	$ -	$ 450	$ 17	$ -	$ -	$ 300	$ -	$ -	$ 1,332
Total Leach Pad Indirects	K USD	$ 2,826	$ -	$ -	$ 2,250	$ 86	$ -	$ -	$ 1,500	$ -	$ -	$ 6,662
Total Pad Construction	K USD	$ 14,130	$ -	$ -	$ 11,250	$ 428	$ -	$ -	$ 7,500	$ -	$ -	$ 33,308
Leach Plant Construction												
Crushing	K USD	$ 22,482	$ -	$ -	$ -	$ 5,347	$ 2,879	$ -	$ -	$ -	$ -	$ 30,707
Agglomeration	K USD	$ -	$ -	$ -	$ -	$ 4,681	$ 2,520	$ -	$ -	$ -	$ -	$ 7,201
Stacking	K USD	$ 5,279	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 5,279
Heap Leach	K USD	$ 741	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 741
Merril-Crowe	K USD	$ 4,101	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,101
Refinery	K USD	$ 1,054	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,054
Reagents	K USD	$ 375	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 375
Utilities	K USD	$ 221	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 221
Misc.	K USD	$ 1,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,000
Power	K USD	$ 1,176	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,176
Subtotal Equipment	K USD	$ 36,428	$ -	$ -	$ -	$ 10,027	$ 5,399	$ -	$ -	$ -	$ -	$ 51,855
Frieght	K USD	$ 1,821	$ -	$ -	$ -	$ 501	$ 270	$ -	$ -	$ -	$ -	$ 2,593
Total Direct Costs	K USD	$ 38,249	$ -	$ -	$ -	$ 10,529	$ 5,669	$ -	$ -	$ -	$ -	$ 54,447
ECPM	K USD	$ 2,550	$ -	$ -	$ -	$ 702	$ 378	$ -	$ -	$ -	$ -	$ 3,630
Contingency	K USD	$ 6,557	$ -	$ -	$ -	$ 1,805	$ 972	$ -	$ -	$ -	$ -	$ 9,334
Owners Costs	K USD	$ 1,093	$ -	$ -	$ -	$ 301	$ 162	$ -	$ -	$ -	$ -	$ 1,556
Total Indirects	K USD	$ 10,200	$ -	$ -	$ -	$ 2,808	$ 1,512	$ -	$ -	$ -	$ -	$ 14,519
Total Plant Construction	K USD	$ 48,449	$ -	$ -	$ -	$ 13,336	$ 7,181	$ -	$ -	$ -	$ -	$ 68,967
Total Leach Pad & Plant Construction												
Directs	K USD	$ 49,553	$ -	$ -	$ 9,000	$ 10,871	$ 5,669	$ -	$ 6,000	$ -	$ -	$ 81,093
Contingency	K USD	$ 8,818	$ -	$ -	$ 1,800	$ 1,873	$ 972	$ -	$ 1,200	$ -	$ -	$ 14,663
ECPM	K USD	$ 3,115	$ -	$ -	$ 450	$ 719	$ 378	$ -	$ 300	$ -	$ -	$ 4,962
Owners Costs	K USD	$ 1,093	$ -	$ -	$ -	$ 301	$ 162	$ -	$ -	$ -	$ -	$ 1,556
Grand Total	K USD	$ 62,579	$ -	$ -	$ 11,250	$ 13,764	$ 7,181	$ -	$ 7,500	$ -	$ -	$ 102,274

Table 21.5 Yearly Florida Mountain Concentrator Processing Capital Costs

Florida Mnt Mill Construction	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Total
Crushing	K USD	$ -	$ 97	$ 4,511	$ 243	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,850
Grinding	K USD	$ -	$ 117	$ 5,425	$ 292	$ -	$ -	$ -	$ -	$ -	$ -	$ 5,834
Flotation	K USD	$ -	$ 44	$ 2,053	$ 110	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,207
Float Con Regrind	K USD	$ -	$ 26	$ 1,212	$ 65	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,303
Conc Leaching	K USD	$ -	$ 44	$ 2,023	$ 109	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,176
CCD	K USD	$ -	$ 35	$ 1,645	$ 88	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,769
Detox	K USD	$ -	$ 84	$ 3,913	$ 210	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,207
Reagents 1	K USD	$ -	$ 9	$ 409	$ 22	$ -	$ -	$ -	$ -	$ -	$ -	$ 439
Reagents 2	K USD	$ -	$ 14	$ 640	$ 34	$ -	$ -	$ -	$ -	$ -	$ -	$ 688
Utilities	K USD	$ -	$ 15	$ 683	$ 37	$ -	$ -	$ -	$ -	$ -	$ -	$ 735
Misc.	K USD	$ -	$ 10	$ 465	$ 25	$ -	$ -	$ -	$ -	$ -	$ -	$ 500
Power	K USD	$ -	$ 22	$ 1,043	$ 56	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,122
Subtotal Equipment	K USD	$ -	$ 517	$ 24,022	$ 1,291	$ -	$ -	$ -	$ -	$ -	$ -	$ 25,830
Frieght	K USD	$ -	$ 26	$ 1,201	$ 65	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,291
Total Direct Costs	K USD	$ -	$ 542	$ 25,223	$ 1,356	$ -	$ -	$ -	$ -	$ -	$ -	$ 27,121
ECPM	K USD	$ -	$ 77	$ 3,603	$ 194	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,874
Contingency	K USD	$ -	$ 52	$ 2,402	$ 129	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,583
Owners Costs	K USD	$ -	$ 15	$ 721	$ 39	$ -	$ -	$ -	$ -	$ -	$ -	$ 775
Total Indirects	K USD	$ -	$ 145	$ 6,726	$ 362	$ -	$ -	$ -	$ -	$ -	$ -	$ 7,232
Total Mill Construction	K USD	$ -	$ 687	$ 31,949	$ 1,718	$ -	$ -	$ -	$ -	$ -	$ -	$ 34,354
Dry Stack Tailings Site Directs	K USD	$ -	$ -	$ 2,350	$ 930	$ -	$ -	$ 2,312	$ -	$ -	$ -	$ 5,592
Contingency	K USD	$ -	$ -	$ 470	$ 186	$ -	$ -	$ 462	$ -	$ -	$ -	$ 1,118
ECPM	K USD	$ -	$ -	$ 118	$ 47	$ -	$ -	$ 116	$ -	$ -	$ -	$ 280
Total Dry Stack Tailings Indirects	K USD	$ -	$ -	$ 588	$ 233	$ -	$ -	$ 578	$ -	$ -	$ -	$ 1,398
Total Dry Stack Tailings	K USD	$ -	$ -	$ 2,938	$ 1,163	$ -	$ -	$ 2,890	$ -	$ -	$ -	$ 6,990
Total Florida Mountain Mill Construction												
Directs	K USD	$ -	$ 542	$ 27,573	$ 2,286	$ -	$ -	$ 2,312	$ -	$ -	$ -	$ 32,713
Contingency	K USD	$ -	$ 52	$ 2,872	$ 315	$ -	$ -	$ 462	$ -	$ -	$ -	$ 3,701
ECPM	K USD	$ -	$ 77	$ 3,721	$ 240	$ -	$ -	$ 116	$ -	$ -	$ -	$ 4,154
Owners Costs	K USD	$ -	$ 15	$ 721	$ 39	$ -	$ -	$ -	$ -	$ -	$ -	$ 775
Grand Total	K USD	$ -	$ 687	$ 34,887	$ 2,880	$ -	$ -	$ 2,890	$ -	$ -	$ -	$ 41,344

21.2.1 Heap-Leach Pad Capital

The heap-leach pad will be constructed in two phases. The first phase will be built over a period of 15 months and will provide approximately one-half of the project heap-leach pad area for the project. The construction period is extended due to seasonal constraints limiting certain outside activities to summer weather conditions. Construction will include foundation preparation, placement of compacted fill for grading the valley side slopes, hauling and placing the clay liner for secondary containment, placement of HDPE material as the primary liner, placement of solution collection piping and permeable liner cover aggregate, and construction of a pump-back system for the pregnant leach solutions from the heap to the Merrill-Crowe recovery system in the plant. The estimated cost for Phase 1 heap-leach pad construction is $14,130,000 including contingency and EPCM. These costs are based on conceptual design quantities and recent costs of similar facilities in Nevada. A 20% contingency was provided in this estimate to allow for changes that may be necessitated by geotechnical investigations and further engineering design. The second half of the heap-leach pad is scheduled to be constructed in year 3. Phase 2 heap-leach pad construction will double the footprint of the containment area and will provide sufficient containment for the LOM heap-leach material. Ongoing capital expenditures for the heap include capital for conveyor modifications, process piping and pumping enhancements, and additions to site infrastructure for placement of leach material. The estimated total sustaining capital cost for year 1 through year 10 is $19,178,000, including 20% contingency and EPCM.

21.2.2 Tailings Impoundment

The estimated cost of the dry stack tailings impoundment is based on placing the filtered tailings material on an HDPE liner with a clay under-liner. Construction will consist of foundation preparation, hauling and placing clay liner, placing HDPE liner, and constructing a meteoric water capture and pump-back system to contain sediment from the deposit. The filtered tailings will be transported to the disposal site with either 40-ton articulated trucks or a conveyor and stacker. The estimated cost of the dry stack tailings impoundment is $6,990,000 including contingency and EPCM. Earthwork and lining costs are based on recent mine construction costs in Nevada.

21.3 Owner and Other Capital Costs

MDA estimated owner and other capital costs to be $33.8 million for the LOM. This includes costs for power, water, access, security, snow removal equipment, warehouse and offices, light vehicles, and preproduction costs. The estimated costs are shown in Table 21.6 with further information as follows:

- Power Distribution – $21.7 million based on discussions with Idaho Power;
- Site Water and Distribution –$150,000 for two water wells and some water distribution piping;
- Access Road – $400,000 for road improvements and access road between deposits (including security and fencing);
- Snow Removal Equipment - $150,000 for a sanding truck with plow;
- Assay lab $2.8 million;
- Buildings and Warehouse - $0.7 million for warehouse and administrative offices;
- Light vehicles - $2.1 million (see Section 21.3.1); and
- Preproduction G&A and Process costs - $5.8 million (see Section 21.3.2). Note: preproduction costs related to mining (pre-stripping) have been included in mining capital (see Section 21.1.6).

Table 21.6 Infrastructure & Owners Capital

Infrastructure & Owner's Cost	Units	Pre-Prod	Yr_5	Total
Power & Substation	K USD	$ 21,714	$ -	$ 21,714
Site Water	K USD	$ 150	$ -	$ 150
Access Road	K USD	$ 300	$ -	$ 300
Security & Fencing	K USD	$ 100	$ -	$ 100
Snow Removal Equipment	K USD	$ 150	$ -	$ 150
Site Warehouse	K USD	$ 200	$ -	$ 200
Administrative Offices	K USD	$ 500	$ -	$ 500
Assay Lab	K USD	$ 2,804	$ -	$ 2,804
Light Vehicles	K USD	$ 1,152	$ 974	$ 2,126
Total Site Infrastructure	K USD	$ 27,070	$ 974	$ 28,044
Preproduction G&A	K USD	$ 4,092	$ -	$ 4,092
Preproduction Process	K USD	$ 1,728	$ -	$ 1,728
Total Preproduction Costs	K USD	$ 5,819	$ -	$ 5,819
Total Infrastructure & Owners	K USD	$ 32,889	$ 974	$ 33,863

21.3.1 Light Vehicles

The light vehicle cost is based on the anticipated vehicle need by department for administration, mining general personnel, mine operations personnel, maintenance, and process personnel. The total cost for light vehicles is estimated to be $2,126,000 as shown in Table 21.7. This estimate includes $1,152,000 in initial costs and $974,000 in replacement costs occurring in year 5.

Table 21.7 Light Vehicle Cost Estimate

Administration	**Units**	**Pre-Prod**	**Yr_5**	**Total**
Project Manager	K USD	$ 39	$ 39	$ 78
Mine General Manager	K USD	$ 37	$ 37	$ 74
Administrative Superintendent / Controller	K USD	$ 37	$ 37	$ 74
Safety and Security Superintendent	K USD	$ 39	$ 39	$ 78
Safety Specialist	K USD	$ 39	$ 39	$ 78
Security Guard	K USD	$ 39	$ 39	$ 78
Environmental Superintendent	K USD	$ 37	$ 37	$ 74
Total Administration	K USD	$ 268	$ 268	$ 536
Mining General Personnel				
Mine Superintendent	K USD	$ 39	$ 39	$ 78
Mine Foremen	K USD	$ 78	$ 78	$ 157
Chief Mine Engineer	K USD	$ 39	$ 39	$ 78
Mine Engineer	K USD	$ -	$ -	$ -
Chief Surveyor	K USD	$ 39	$ 39	$ 78
Chief Geologist	K USD	$ 39	$ 39	$ 78
Samplers	K USD	$ 39	$ 39	$ 78
Total Mine General	K USD	$ 275	$ 275	$ 549
Mine Operations Hourly Personnel				
Operators - In Pit Vans	K USD	$ 178	$ -	$ 178
Blasters	K USD	$ 39	$ 39	$ 78
Total Mine Operations	K USD	$ 217	$ 39	$ 257
Maintenance				
Maintenance Foreman	K USD	$ 78	$ 78	$ 157
Heavy Equipment Mechanic	K USD	$ 39	$ 39	$ 78
Total Maintenance	K USD	$ 118	$ 118	$ 235
Process				
Process Manager	K USD	$ 39	$ 39	$ 78
Process General Foreman	K USD	$ 39	$ 39	$ 78
Shift Foreman	K USD	$ 39	$ 39	$ 78
Maintenance Foreman	K USD	$ 39	$ 39	$ 78
Heap Leach Crew	K USD	$ 39	$ 39	$ 78
Maintenance Crew	K USD	$ 39	$ 39	$ 78
Lab	K USD	$ 39	$ 39	$ 78
Process Total	K USD	$ 275	$ 275	$ 549
Total	K USD	$ 1,152	$ 974	$ 2,126

21.3.2 Preproduction Owner Costs

Preproduction owner's costs include general and administrative ("G&A"), and some processing costs during preproduction. Preproduction costs related to mining (pre-stripping) have been included in mining capital (see Section 21.1.6).

The G&A costs include construction management personnel as well as staff for administration, accounting, environmental, and safety and security functions. The total preproduction G&A cost is estimated to be $4,092,000.

Preproduction process costs are estimated to be $1,728,000. This is estimated using fixed and variable costs for tonnage of material sent to the Florida Mountain crusher and leach pad during two months of preproduction. It is assumed that some of this material will be placed on the pad as over-liner material and the remaining of the material will be used for commissioning.

21.4 Reclamation Costs and Salvage Value

Reclamation costs were estimated to be approximately $20 million at the end of the mine life. The reclamation costs were estimated using the Standardized Reclamation Cost Estimator (version 1.4) ("RCE"). The RCE has been developed in accordance with the guidelines created by Nevada Standardized Unit Cost Project, a cooperative effort of the NDEP, the U.S. Department of Interior, Bureau of Land Management, and the Nevada Mining Association.

These costs are estimated based on project specific data. Direct costs of $14.2 million were estimated with additional $5.0 million in indirect costs. The total estimate comes to $19.1 million which was rounded to $20.0 million for the PEA.

The reclamation costs were assumed to be secured with a surety bond. A required cash collateral deposit of $6.0 million (30%) was assumed for the surety bond and included in pre-production capital. The surety bond yearly fees were allocated to G&A operating costs.

A credit of 6% for process equipment will be taken as a salvage value at the end of processing. This amounts to $2.9 million in year 11 for process equipment related to leaching, and $1.1 million in year 10 for milling equipment.

A total of $22.4 million in salvage was credited to capital accounts for primary and support mining equipment. This was estimated using the initial cost basis reduced by 20% for initial depreciation, then further reduced based on anticipated life of equipment in terms of hours by equipment type. The mining equipment salvage credit will be taken in year 11.

21.5 Mine Operating Costs

Mine operating costs were estimated using first principals. This was done using estimated hourly costs of equipment and personnel against the anticipated hours of work for each. The equipment hourly costs were estimated for fuel, oil and lubrication, tires, under-carriage, repair and maintenance costs, and special wear items.

Personnel costs include supervision, operating labor, and maintenance labor. The mine operating costs are summarized by year in Table 21.8. Note that while the costs for preproduction are shown in the cost tables below, these costs are capitalized as preproduction costs. After capitalization of preproduction costs, the LOM mining costs are $392.2 million or $2.00/t mined.

Table 21.8 Yearly Mine Operating Cost Estimate

Mine Op Cost Summary	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Mine General Service	K USD	$ 445	$ 844	$ 844	$ 844	$ 844	$ 844	$ 844	$ 844	$ 844	$ 844	$ 320	$ -	$ 8,359
Mine Maintenance	K USD	$ 1,266	$ 2,526	$ 2,532	$ 2,533	$ 2,532	$ 2,532	$ 2,532	$ 2,533	$ 2,532	$ 2,532	$ 1,033	$ -	$ 25,084
Engineering	K USD	$ 424	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 303	$ -	$ 7,871
Geology	K USD	$ 464	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 349	$ -	$ 8,801
Drilling	K USD	$ 650	$ 4,067	$ 5,083	$ 6,671	$ 7,192	$ 5,826	$ 4,018	$ 3,938	$ 3,978	$ 3,483	$ 980	$ -	$ 45,887
Blasting	K USD	$ 495	$ 3,807	$ 4,661	$ 6,054	$ 6,649	$ 5,243	$ 3,745	$ 3,705	$ 3,695	$ 3,315	$ 934	$ -	$ 42,304
Loading	K USD	$ 640	$ 4,355	$ 5,487	$ 7,335	$ 7,750	$ 6,046	$ 4,299	$ 4,351	$ 4,225	$ 3,697	$ 1,086	$ -	$ 49,270
Hauling	K USD	$ 1,305	$ 14,917	$ 18,065	$ 22,607	$ 24,059	$ 19,545	$ 14,707	$ 12,516	$ 13,618	$ 13,299	$ 4,266	$ -	$ 158,902
Mine Support	K USD	$ 1,824	$ 5,463	$ 5,463	$ 5,472	$ 5,463	$ 5,463	$ 5,463	$ 5,472	$ 5,463	$ 5,463	$ 2,267	$ -	$ 53,279
Total Mining Cost	K USD	$ 7,514	$ 37,661	$ 43,817	$ 53,197	$ 56,170	$ 47,180	$ 37,290	$ 35,039	$ 36,036	$ 34,314	$ 11,538	$ -	$ 399,757
Total After Capitalization	K USD	$ -	$ 37,661	$ 43,817	$ 53,197	$ 56,170	$ 47,180	$ 37,290	$ 35,039	$ 36,036	$ 34,314	$ 11,538	$ -	$ 392,243
Cost per Tonne														
Mine General Service	$/t	$ 0.32	$ 0.05	$ 0.04	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.08	$ -	$ 0.04
Mine Maintenance	$/t	$ 0.92	$ 0.14	$ 0.12	$ 0.09	$ 0.08	$ 0.10	$ 0.15	$ 0.15	$ 0.15	$ 0.17	$ 0.27	$ -	$ 0.13
Engineering	$/t	$ 0.31	$ 0.05	$ 0.04	$ 0.03	$ 0.02	$ 0.03	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.08	$ -	$ 0.04
Geology	$/t	$ 0.34	$ 0.05	$ 0.04	$ 0.03	$ 0.03	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.09	$ -	$ 0.04
Drilling	$/t	$ 0.47	$ 0.23	$ 0.23	$ 0.23	$ 0.22	$ 0.24	$ 0.23	$ 0.23	$ 0.24	$ 0.23	$ 0.25	$ -	$ 0.23
Blasting	$/t	$ 0.36	$ 0.22	$ 0.21	$ 0.21	$ 0.21	$ 0.22	$ 0.22	$ 0.22	$ 0.22	$ 0.22	$ 0.24	$ -	$ 0.22
Loading	$/t	$ 0.47	$ 0.25	$ 0.25	$ 0.25	$ 0.24	$ 0.25	$ 0.25	$ 0.26	$ 0.25	$ 0.25	$ 0.28	$ -	$ 0.25
Hauling	$/t	$ 0.95	$ 0.85	$ 0.82	$ 0.78	$ 0.75	$ 0.80	$ 0.86	$ 0.74	$ 0.81	$ 0.89	$ 1.10	$ -	$ 0.81
Mine Support	$/t	$ 1.33	$ 0.31	$ 0.25	$ 0.19	$ 0.17	$ 0.22	$ 0.32	$ 0.32	$ 0.32	$ 0.37	$ 0.58	$ -	$ 0.27
Total Mining Cost	$/t	$ 5.48	$ 2.16	$ 2.00	$ 1.83	$ 1.74	$ 1.94	$ 2.18	$ 2.07	$ 2.14	$ 2.30	$ 2.97	$ -	$ 2.04
Total After Capitalization	$/t	$ -	$ 2.16	$ 2.00	$ 1.83	$ 1.74	$ 1.94	$ 2.18	$ 2.07	$ 2.14	$ 2.30	$ 2.97	$ -	$ 2.00

21.5.1 Mine General Services

Mine general services includes mining supervision along with engineering and geology services. Supervision allows for a mine superintendent, mine general foreman and mine shift foremen. Engineering personnel include a chief engineer along with engineers and surveying crew to support mine planning and operations. Geology is intended to support ore control, geological mapping, and sampling requirements.

Table 21.9 shows the yearly cost estimate for the mine general services.

Table 21.9 Mine General Services, Engineering and Geology Costs

Mine General Services	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Supervision	K USD	$ 367	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 253	$ -	$ 6,706
Hourly Personnel	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total	K USD	$ 367	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 253	$ -	$ 6,706
Engineering														
Salaried Personnel	K USD	$ 248	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 156	$ -	$ 4,379
Hourly Personnel	K USD	$ 161	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 134	$ -	$ 3,191
Total	K USD	$ 409	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 290	$ -	$ 7,570
Mine Geology														
Salaried Personnel	K USD	$ 83	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 31	$ -	$ 1,232
Hourly Personnel	K USD	$ 363	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 303	$ -	$ 7,202
Total	K USD	$ 446	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 334	$ -	$ 8,434
Supplies & Other														
Mine General Services Supplies	K USD	$ 6	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 5	$ -	$ 121
Engineering Supplies	K USD	$ 15	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 13	$ -	$ 301
Geology Supplies	K USD	$ 19	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 15	$ -	$ 367
Software Maintanance & Support	K USD	$ 15	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 12	$ -	$ 288
Outside Services	K USD	$ 38	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 31	$ -	$ 744
Office Power	K USD	$ 5	$ 11	$ 11	$ 11	$ 11	$ 11	$ 11	$ 11	$ 11	$ 11	$ 5	$ -	$ 109
Light Vehicles	K USD	$ 29	$ 69	$ 69	$ 70	$ 69	$ 69	$ 69	$ 70	$ 69	$ 69	$ 26	$ -	$ 680
Total	K USD	$ 126	$ 264	$ 264	$ 264	$ 264	$ 264	$ 264	$ 264	$ 264	$ 264	$ 107	$ -	$ 2,609
Totals - Mining General														
Mine General	K USD	$ 445	$ 844	$ 844	$ 844	$ 844	$ 844	$ 844	$ 844	$ 844	$ 844	$ 320	$ -	$ 8,359
Engineering	K USD	$ 424	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 303	$ -	$ 7,871
Geology	K USD	$ 464	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 349	$ -	$ 8,801
Totals	K USD	$ 1,334	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 972	$ -	$ 25,031
Total After Capitalization	K USD	$ -	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 2,525	$ 972	$ -	$ 23,697
Cost per Tonne Mined														
Mine General	$/t	$ 0.32	$ 0.05	$ 0.04	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.08	$ -	$ 0.04
Engineering	$/t	$ 0.31	$ 0.05	$ 0.04	$ 0.03	$ 0.02	$ 0.03	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.08	$ -	$ 0.04
Geology	$/t	$ 0.34	$ 0.05	$ 0.04	$ 0.03	$ 0.03	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.09	$ -	$ 0.04
Totals	$/t	$ 0.97	$ 0.14	$ 0.12	$ 0.09	$ 0.08	$ 0.10	$ 0.15	$ 0.15	$ 0.15	$ 0.17	$ 0.25	$ -	$ 0.13
Total After Capitalization	$/t	$ -	$ 0.14	$ 0.12	$ 0.09	$ 0.08	$ 0.10	$ 0.15	$ 0.15	$ 0.15	$ 0.17	$ 0.25	$ -	$ 0.12

21.5.2 Mine Maintenance

Mine maintenance costs include the cost of personnel for maintenance, supervision, and planning, along with shop support personnel, including light vehicle mechanics, welders, servicemen, tire men, and maintenance labor.

The estimated mine maintenance costs are shown in Table 21.10. Note that these costs do not include the maintenance labor directly allocated to the various equipment, which is accounted for in the other mining cost categories.

Table 21.10 Yearly Mine Maintenance Costs

Wages & Salaries	**Units**	**Pre-Prod**	**Yr_1**	**Yr_2**	**Yr_3**	**Yr_4**	**Yr_5**	**Yr_6**	**Yr_7**	**Yr_8**	**Yr_9**	**Yr_10**	**Yr_11**	**Total**
Supervision	K USD	$ 276	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 192	$ -	$ 5,063
Planners	K USD	$ 77	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 64	$ -	$ 1,534
Hourly Personnel	K USD	$ 662	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 552	$ -	$ 13,131
Total	K USD	$ 1,015	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 808	$ -	$ 19,729
Other Costs														
Supplies	K USD	$ 72	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 60	$ -	$ 1,428
Light Vehicles	K USD	$ 9	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 9	$ -	$ 210
Total	K USD	$ 81	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 69	$ -	$ 1,638
Consumables & Other Costs	K USD	$ 214	$ 458	$ 464	$ 465	$ 464	$ 464	$ 464	$ 465	$ 464	$ 464	$ 192	$ -	$ 4,578
Parts / MARC Cost	K USD	$ 36	$ 78	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 33	$ -	$ 777
Wages & Salaries	K USD	$ 1,015	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 808	$ -	$ 19,729
Total	K USD	$ 1,266	$ 2,526	$ 2,532	$ 2,533	$ 2,532	$ 2,532	$ 2,532	$ 2,533	$ 2,532	$ 2,532	$ 1,033	$ -	$ 25,084
Total After Capitalization	K USD	$ -	$ 2,526	$ 2,532	$ 2,533	$ 2,532	$ 2,532	$ 2,532	$ 2,533	$ 2,532	$ 2,532	$ 1,033	$ -	$ 23,818
Consumables	$/t	$ 0.16	$ 0.03	$ 0.02	$ 0.02	$ 0.01	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ -	$ 0.02
Parts / MARC Cost	$/t	$ 0.03	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.01	$ 0.01	$ -	$ 0.00
Maintenance Labor	$/t	$ 0.74	$ 0.11	$ 0.09	$ 0.07	$ 0.06	$ 0.08	$ 0.12	$ 0.12	$ 0.12	$ 0.13	$ 0.21	$ -	$ 0.10
Total	$/t	$ 0.92	$ 0.14	$ 0.12	$ 0.09	$ 0.08	$ 0.10	$ 0.15	$ 0.15	$ 0.15	$ 0.17	$ 0.27	$ -	$ 0.13
Total After Capitalization	$/t	$ -	$ 0.14	$ 0.12	$ 0.09	$ 0.08	$ 0.10	$ 0.15	$ 0.15	$ 0.15	$ 0.17	$ 0.27	$ -	$ 0.12

21.5.3 Drilling

Drilling cost estimates are shown in Table 21.11. The LOM drilling costs are estimated to be $45.9 million or $0.23 per tonne including preproduction. The total LOM cost without capitalized preproduction is $45.2 million.

Table 21.11 Yearly Drilling Costs

Drilling Operating Cost	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Prod Drill Fuel Consumption	K Liters	67	896	1,124	1,496	1,655	1,199	880	869	866	765	199	-	10,017
Prod Drill Fuel Cost	K USD	$ 44	$ 592	$ 743	$ 988	$ 1,093	$ 792	$ 581	$ 574	$ 572	$ 505	$ 132	$ -	$ 6,616
Prod Drill Lube & Oil	K USD	$ 21	$ 287	$ 360	$ 479	$ 530	$ 384	$ 282	$ 278	$ 278	$ 245	$ 64	$ -	$ 3,209
Prod Drill Undercarriage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Prod Drill Drill Bits & Steel	K USD	$ 67	$ 891	$ 1,117	$ 1,486	$ 1,644	$ 1,191	$ 874	$ 863	$ 861	$ 760	$ 198	$ -	$ 9,953
Prod Drill Total Consumables	K USD	$ 132	$ 1,770	$ 2,220	$ 2,953	$ 3,267	$ 2,367	$ 1,737	$ 1,716	$ 1,711	$ 1,510	$ 394	$ -	$ 19,777
Prod Drill Parts	K USD	$ 67	$ 891	$ 1,117	$ 1,486	$ 1,644	$ 1,191	$ 874	$ 863	$ 861	$ 760	$ 198	$ -	$ 9,953
Prod Drill Maintenance Labor	K USD	$ 107	$ 456	$ 566	$ 724	$ 739	$ 587	$ 456	$ 441	$ 456	$ 393	$ 126	$ -	$ 5,053
Pioneer Drill Fuel Consumption	K Liters	8	-	-	-	-	124	-	-	-	-	-	-	131
Pioneer Drill Fuel Cost	K USD	$ 5	$ -	$ -	$ -	$ -	$ 82	$ -	$ -	$ -	$ -	$ -	$ -	$ 87
Pioneer Drill Lube & Oil	K USD	$ 1	$ -	$ -	$ -	$ -	$ 23	$ -	$ -	$ -	$ -	$ -	$ -	$ 25
Pioneer Drill Undercarriage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pioneer Drill Drill Bits & Steel	K USD	$ 4	$ -	$ -	$ -	$ -	$ 62	$ -	$ -	$ -	$ -	$ -	$ -	$ 66
Pioneer Drill Total Consumables	K USD	$ 11	$ -	$ -	$ -	$ -	$ 167	$ -	$ -	$ -	$ -	$ -	$ -	$ 178
Pioneer Drill Parts / MARC Cost	K USD	$ 4	$ -	$ -	$ -	$ -	$ 62	$ -	$ -	$ -	$ -	$ -	$ -	$ 66
Pioneer Drill Maintenance Labor	K USD	$ 34	$ -	$ -	$ -	$ -	$ 74	$ -	$ -	$ -	$ -	$ -	$ -	$ 108
Total Drill Fuel Consumption	K Liters	75	896	1,124	1,496	1,655	1,323	880	869	866	765	199	-	10,149
Total Drill Fuel Cost	K USD	$ 50	$ 592	$ 743	$ 988	$ 1,093	$ 873	$ 581	$ 574	$ 572	$ 505	$ 132	$ -	$ 6,703
Total Drill Lube & Oil	K USD	$ 23	$ 287	$ 360	$ 479	$ 530	$ 407	$ 282	$ 278	$ 278	$ 245	$ 64	$ -	$ 3,234
Total Drill Undercarriage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Drill Drill Bits & Steel	K USD	$ 71	$ 891	$ 1,117	$ 1,486	$ 1,644	$ 1,254	$ 874	$ 863	$ 861	$ 760	$ 198	$ -	$ 10,019
Total Drill Total Consumables	K USD	$ 143	$ 1,770	$ 2,220	$ 2,953	$ 3,267	$ 2,534	$ 1,737	$ 1,716	$ 1,711	$ 1,510	$ 394	$ -	$ 19,955
Total Drill Parts / MARC Cost	K USD	$ 71	$ 891	$ 1,117	$ 1,486	$ 1,644	$ 1,254	$ 874	$ 863	$ 861	$ 760	$ 198	$ -	$ 10,019
Total Drill Maintenance Labor	K USD	$ 142	$ 456	$ 566	$ 724	$ 739	$ 661	$ 456	$ 441	$ 456	$ 393	$ 126	$ -	$ 5,161
Total Drill Total Maintenance Allocation	K USD	$ 212	$ 1,347	$ 1,684	$ 2,210	$ 2,384	$ 1,914	$ 1,330	$ 1,304	$ 1,317	$ 1,153	$ 324	$ -	$ 15,180
Total Operator Wages & Burden	K USD	$ 295	$ 951	$ 1,180	$ 1,508	$ 1,541	$ 1,377	$ 951	$ 918	$ 951	$ 820	$ 262	$ -	$ 10,752
Total Drilling Cost	K USD	$ 650	$ 4,067	$ 5,083	$ 6,671	$ 7,192	$ 5,826	$ 4,018	$ 3,938	$ 3,978	$ 3,483	$ 980	$ -	$ 45,887
Total After Capitalization	K USD	$ -	$ 4,067	$ 5,083	$ 6,671	$ 7,192	$ 5,826	$ 4,018	$ 3,938	$ 3,978	$ 3,483	$ 980	$ -	$ 45,236
Drilling Cost per Tonne Mined by Item														
Fuel Cost	$/t	$ 0.04	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ -	$ 0.03
Lube & Oil	$/t	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ -	$ 0.02
Undercarriage	$/t	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Drill Bits & Steel	$/t	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ -	$ 0.05
Total Consumables	$/t	$ 0.10	$ 0.10	$ 0.10	$ 0.10	$ 0.10	$ 0.10	$ 0.10	$ 0.10	$ 0.10	$ 0.10	$ 0.10	$ -	$ 0.10
Parts / MARC Cost	$/t	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ -	$ 0.05
Maintenance Labor	$/t	$ 0.10	$ 0.03	$ 0.03	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ -	$ 0.03
Total Maintenance Allocation	$/t	$ 0.15	$ 0.08	$ 0.08	$ 0.08	$ 0.07	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ -	$ 0.08
Operator Wages & Burden	$/t	$ 0.22	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.06	$ 0.05	$ 0.06	$ 0.06	$ 0.07	$ -	$ 0.05
Total Drilling Cost	$/t	$ 0.47	$ 0.23	$ 0.23	$ 0.23	$ 0.22	$ 0.24	$ 0.23	$ 0.23	$ 0.24	$ 0.23	$ 0.25	$ -	$ 0.23
Total After Capitalization	$/t	$ -	$ 0.23	$ 0.23	$ 0.23	$ 0.22	$ 0.24	$ 0.23	$ 0.23	$ 0.24	$ 0.23	$ 0.25	$ -	$ 0.23

21.5.4 Blasting

LOM blasting costs, including preproduction, are shown in Table 21.12. These costs are based on owner operations for blasting and assume ANFO costs of $600/tonne with transportation costs for ANFO at $35/tonne. A blasting accessories cost of $22.00 per hole was included.

Table 21.12 Yearly Blasting Costs

Blasting Costs	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Fuel	K Liters	47	93	93	93	93	93	93	93	93	93	38	-	921
Blasting Consumables	K USD	$ 271	$ 3,359	$ 4,213	$ 5,605	$ 6,201	$ 4,795	$ 3,297	$ 3,256	$ 3,247	$ 2,866	$ 747	$ -	$ 37,856
Equipment Consumables	K USD	$ 36	$ 71	$ 71	$ 71	$ 71	$ 71	$ 71	$ 71	$ 71	$ 71	$ 29	$ -	$ 702
Equipment Maintenance Allocations	K USD	$ 8	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 6	$ -	$ 153
Personnel	K USD	$ 169	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 141	$ -	$ 3,355
Supplies	K USD	$ 6	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 5	$ -	$ 119
Outside Services	K USD	$ 6	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 5	$ -	$ 119
Total Blasting Costs	K USD	$ 495	$ 3,807	$ 4,661	$ 6,054	$ 6,649	$ 5,243	$ 3,745	$ 3,705	$ 3,695	$ 3,315	$ 934	$ -	$ 42,304
Total After Capitalization	K USD	$ -	$ 3,807	$ 4,661	$ 6,054	$ 6,649	$ 5,243	$ 3,745	$ 3,705	$ 3,695	$ 3,315	$ 934	$ -	$ 41,809
Cost per Tonne														
Blasting Consumables	$/t	$ 0.20	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.20	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ -	$ 0.19
Equipment Consumables	$/t	$ 0.03	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.01	$ -	$ 0.00
Equipment Maintenance Allocations	$/t	$ 0.01	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ -	$ 0.00
Personnel	$/t	$ 0.12	$ 0.02	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.04	$ -	$ 0.02
Supplies	$/t	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ -	$ 0.00
Outside Services	$/t	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ -	$ 0.00
Total	$/t	$ 0.36	$ 0.22	$ 0.21	$ 0.21	$ 0.21	$ 0.22	$ 0.22	$ 0.22	$ 0.22	$ 0.22	$ 0.24	$ -	$ 0.22
Total After Capitalization	$/t	$ -	$ 0.22	$ 0.21	$ 0.21	$ 0.21	$ 0.22	$ 0.22	$ 0.22	$ 0.22	$ 0.22	$ 0.24	$ -	$ 0.21

21.5.5 Loading

Loading costs are based on operation of two hydraulic shovels with 22 cubic meter buckets for all primary production. In addition, a 21 cubic yard front-end-loader is assumed to be used for stockpile management and re-handling as well as backup for production during shovel maintenance. The yearly loading cost estimate is shown in Table 21.13.

Table 21.13 Yearly Loading Costs

Shovel Costs	**Units**	**Pre-Prod**	**Yr_1**	**Yr_2**	**Yr_3**	**Yr_4**	**Yr_5**	**Yr_6**	**Yr_7**	**Yr_8**	**Yr_9**	**Yr_10**	**Yr_11**	**Total**
Fuel Consumption	K Liters	52	622	792	1,108	1,021	741	521	515	513	453	118	-	6,458
Fuel Cost	K USD	$ 34	$ 411	$ 523	$ 732	$ 674	$ 490	$ 344	$ 340	$ 339	$ 299	$ 78	$ -	$ 4,265
Lube & Oil	K USD	$ 32	$ 380	$ 483	$ 676	$ 623	$ 452	$ 318	$ 314	$ 313	$ 277	$ 72	$ -	$ 3,940
Tires / Under Carriage	K USD	$ 22	$ 264	$ 336	$ 470	$ 433	$ 315	$ 221	$ 219	$ 218	$ 192	$ 50	$ -	$ 2,742
Wear Items & GET	K USD	$ 26	$ 305	$ 388	$ 543	$ 500	$ 363	$ 255	$ 252	$ 252	$ 222	$ 58	$ -	$ 3,163
Total Consumables	K USD	$ 114	$ 1,360	$ 1,730	$ 2,421	$ 2,231	$ 1,620	$ 1,139	$ 1,125	$ 1,122	$ 990	$ 258	$ -	$ 14,110
Parts / MARC Cost	K USD	$ 145	$ 1,736	$ 2,209	$ 3,091	$ 2,848	$ 2,068	$ 1,454	$ 1,437	$ 1,432	$ 1,265	$ 330	$ -	$ 18,016
Total Equip. Allocation (no labor)	K USD	$ 259	$ 3,096	$ 3,939	$ 5,512	$ 5,079	$ 3,688	$ 2,594	$ 2,562	$ 2,554	$ 2,255	$ 588	$ -	$ 32,126
Loader Cost														
Fuel Consumption	K Liters	25	154	159	65	689	667	456	559	456	342	127	-	3,698
Fuel Cost	K USD	$ 17	$ 102	$ 105	$ 43	$ 455	$ 441	$ 301	$ 369	$ 301	$ 226	$ 84	$ -	$ 2,442
Lube & Oil	K USD	$ 5	$ 32	$ 33	$ 13	$ 142	$ 138	$ 94	$ 115	$ 94	$ 70	$ 26	$ -	$ 762
Tires / Under Carriage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wear Items & GET	K USD	$ 2	$ 10	$ 10	$ 4	$ 45	$ 44	$ 30	$ 37	$ 30	$ 22	$ 8	$ -	$ 242
Total Consumables	K USD	$ 23	$ 143	$ 148	$ 60	$ 642	$ 622	$ 425	$ 521	$ 425	$ 318	$ 118	$ -	$ 3,446
Parts / MARC Cost	K USD	$ 9	$ 57	$ 59	$ 24	$ 257	$ 249	$ 170	$ 208	$ 170	$ 128	$ 47	$ -	$ 1,380
Total Equip. Allocation (no labor)	K USD	$ 33	$ 201	$ 208	$ 84	$ 899	$ 871	$ 596	$ 729	$ 595	$ 446	$ 165	$ -	$ 4,826
Total Loading Cost														
Fuel Consumption	K Liters	77	776	951	1,173	1,710	1,409	978	1,074	969	795	245	-	10,156
Fuel Cost	K USD	$ 51	$ 513	$ 628	$ 774	$ 1,129	$ 930	$ 646	$ 709	$ 640	$ 525	$ 162	$ -	$ 6,707
Lube & Oil	K USD	$ 37	$ 411	$ 516	$ 689	$ 765	$ 590	$ 412	$ 429	$ 407	$ 347	$ 98	$ -	$ 4,702
Tires / Under Carriage	K USD	$ 22	$ 264	$ 336	$ 470	$ 433	$ 315	$ 221	$ 219	$ 218	$ 192	$ 50	$ -	$ 2,742
Wear Items & GET	K USD	$ 27	$ 315	$ 398	$ 547	$ 545	$ 407	$ 285	$ 289	$ 281	$ 244	$ 66	$ -	$ 3,405
Total Consumables	K USD	$ 137	$ 1,503	$ 1,878	$ 2,481	$ 2,873	$ 2,242	$ 1,564	$ 1,646	$ 1,546	$ 1,309	$ 376	$ -	$ 17,556
Parts / MARC Cost	K USD	$ 155	$ 1,794	$ 2,268	$ 3,115	$ 3,105	$ 2,317	$ 1,625	$ 1,645	$ 1,602	$ 1,392	$ 377	$ -	$ 19,396
Total Equip. Allocation (no labor)	K USD	$ 292	$ 3,297	$ 4,146	$ 5,596	$ 5,978	$ 4,559	$ 3,189	$ 3,291	$ 3,149	$ 2,701	$ 753	$ -	$ 36,952
Maintenance Labor	K USD	$ 142	$ 456	$ 566	$ 724	$ 739	$ 661	$ 456	$ 441	$ 456	$ 393	$ 126	$ -	$ 5,161
Operator Wages & Burden	K USD	$ 206	$ 602	$ 774	$ 1,015	$ 1,032	$ 826	$ 654	$ 619	$ 619	$ 602	$ 206	$ -	$ 7,158
Total Loading Costs	K USD	$ 640	$ 4,355	$ 5,487	$ 7,335	$ 7,750	$ 6,046	$ 4,299	$ 4,351	$ 4,225	$ 3,697	$ 1,086	$ -	$ 49,270
Total After Capitalization	K USD	$ -	$ 4,355	$ 5,487	$ 7,335	$ 7,750	$ 6,046	$ 4,299	$ 4,351	$ 4,225	$ 3,697	$ 1,086	$ -	$ 48,630
Cost per Tonne														
Fuel Cost	$/t	$ 0.04	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ -	$ 0.03
Lube & Oil	$/t	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.02	$ 0.02	$ 0.03	$ -	$ 0.02
Tires / Under Carriage	$/t	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ -	$ 0.01
Wear Items & GET	$/t	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ -	$ 0.02
Total Consumables	$/t	$ 0.10	$ 0.09	$ 0.09	$ 0.09	$ 0.09	$ 0.09	$ 0.09	$ 0.10	$ 0.09	$ 0.09	$ 0.10	$ -	$ 0.09
Parts / MARC Cost	$/t	$ 0.11	$ 0.10	$ 0.10	$ 0.11	$ 0.10	$ 0.10	$ 0.09	$ 0.10	$ 0.09	$ 0.09	$ 0.10	$ -	$ 0.10
Total Equip. Allocation (no labor)	$/t	$ 0.21	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.18	$ 0.19	$ -	$ 0.19
Maintenance Labor	$/t	$ 0.10	$ 0.03	$ 0.03	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ -	$ 0.03
Operator Wages & Burden	$/t	$ 0.15	$ 0.03	$ 0.04	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.05	$ -	$ 0.04
Total Loading Cost	$/t	$ 0.47	$ 0.25	$ 0.25	$ 0.25	$ 0.24	$ 0.25	$ 0.25	$ 0.26	$ 0.25	$ 0.25	$ 0.28	$ -	$ 0.25
Total After Capitalization	$/t	$ -	$ 0.25	$ 0.25	$ 0.25	$ 0.24	$ 0.25	$ 0.25	$ 0.26	$ 0.25	$ 0.25	$ 0.28	$ -	$ 0.25

21.5.6 Hauling

Haulage cost was estimated using the truck hour estimates discussed in Section 16.10. The yearly haulage cost estimate is shown in Table 21.14.

Table 21.14 Yearly Haulage Costs

Haulage Costs	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Fuel Consumption	K Liters	467	6,765	8,191	10,368	10,520	8,298	6,304	5,397	6,142	5,926	1,607	-	69,985
Fuel Cost	K USD	$ 308	$ 4,468	$ 5,409	$ 6,847	$ 6,948	$ 5,480	$ 4,163	$ 3,564	$ 4,056	$ 3,914	$ 1,062	$ -	$ 46,220
Lube & Oil	K USD	$ 104	$ 1,509	$ 1,827	$ 2,312	$ 2,346	$ 1,851	$ 1,406	$ 1,204	$ 1,370	$ 1,322	$ 359	$ -	$ 15,610
Tires	K USD	$ 142	$ 2,063	$ 2,498	$ 3,162	$ 3,209	$ 2,531	$ 1,923	$ 1,646	$ 1,873	$ 1,808	$ 490	$ -	$ 21,346
Wear Items & GET	K USD	$ 11	$ 161	$ 195	$ 247	$ 251	$ 198	$ 150	$ 129	$ 146	$ 141	$ 38	$ -	$ 1,668
Total Consumables	K USD	$ 566	$ 8,201	$ 9,930	$ 12,569	$ 12,753	$ 10,060	$ 7,642	$ 6,543	$ 7,446	$ 7,185	$ 1,949	$ -	$ 84,844
Parts / MARC Cost	K USD	$ 125	$ 1,807	$ 2,188	$ 2,769	$ 2,810	$ 2,216	$ 1,684	$ 1,441	$ 1,640	$ 1,583	$ 429	$ -	$ 18,691
Total Equip. Allocation (no labor)	K USD	$ 691	$ 10,008	$ 12,117	$ 15,338	$ 15,563	$ 12,276	$ 9,326	$ 7,984	$ 9,086	$ 8,768	$ 2,378	$ -	$ 103,535
Maintenance Labor	K USD	$ 205	$ 1,636	$ 1,982	$ 2,423	$ 2,832	$ 2,423	$ 1,794	$ 1,510	$ 1,510	$ 1,510	$ 629	$ -	$ 18,456
Operator Wages & Burden	K USD	$ 409	$ 3,273	$ 3,965	$ 4,846	$ 5,664	$ 4,846	$ 3,587	$ 3,021	$ 3,021	$ 3,021	$ 1,259	$ -	$ 36,911
Total Haulage Costs	K USD	$ 1,305	$ 14,917	$ 18,065	$ 22,607	$ 24,059	$ 19,545	$ 14,707	$ 12,516	$ 13,618	$ 13,299	$ 4,266	$ -	$ 158,902
Total After Capitalization	K USD	$ -	$ 14,917	$ 18,065	$ 22,607	$ 24,059	$ 19,545	$ 14,707	$ 12,516	$ 13,618	$ 13,299	$ 4,266	$ -	$ 157,597
Cost per Tonne Mined														
Fuel Cost	$/t	$ 0.22	$ 0.26	$ 0.25	$ 0.24	$ 0.22	$ 0.22	$ 0.24	$ 0.21	$ 0.24	$ 0.26	$ 0.27	$ -	$ 0.24
Lube & Oil	$/t	$ 0.08	$ 0.09	$ 0.08	$ 0.08	$ 0.07	$ 0.08	$ 0.08	$ 0.07	$ 0.08	$ 0.09	$ 0.09	$ -	$ 0.08
Tires	$/t	$ 0.10	$ 0.12	$ 0.11	$ 0.11	$ 0.10	$ 0.10	$ 0.11	$ 0.10	$ 0.11	$ 0.12	$ 0.13	$ -	$ 0.11
Wear Items & GET	$/t	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ -	$ 0.01
Total Consumables	$/t	$ 0.41	$ 0.47	$ 0.45	$ 0.43	$ 0.40	$ 0.41	$ 0.45	$ 0.39	$ 0.44	$ 0.48	$ 0.50	$ -	$ 0.43
Parts / MARC Cost	$/t	$ 0.09	$ 0.10	$ 0.10	$ 0.10	$ 0.09	$ 0.09	$ 0.10	$ 0.09	$ 0.10	$ 0.11	$ 0.11	$ -	$ 0.10
Total Equip. Allocation (no labor)	$/t	$ 0.50	$ 0.57	$ 0.55	$ 0.53	$ 0.48	$ 0.50	$ 0.54	$ 0.47	$ 0.54	$ 0.59	$ 0.61	$ -	$ 0.53
Maintenance Labor	$/t	$ 0.15	$ 0.09	$ 0.09	$ 0.08	$ 0.09	$ 0.10	$ 0.10	$ 0.09	$ 0.09	$ 0.10	$ 0.16	$ -	$ 0.09
Operator Wages & Burden	$/t	$ 0.30	$ 0.19	$ 0.18	$ 0.17	$ 0.18	$ 0.20	$ 0.21	$ 0.18	$ 0.18	$ 0.20	$ 0.32	$ -	$ 0.19
Total Haulage Costs	$/t	$ 0.95	$ 0.85	$ 0.82	$ 0.78	$ 0.75	$ 0.80	$ 0.86	$ 0.74	$ 0.81	$ 0.89	$ 1.10	$ -	$ 0.81
Total After Capitalization	$/t	$ -	$ 0.85	$ 0.82	$ 0.78	$ 0.75	$ 0.80	$ 0.86	$ 0.74	$ 0.81	$ 0.89	$ 1.10	$ -	$ 0.80

21.5.7 Mine Support

Yearly mine support cost estimates are shown in Table 21.15 including preproduction costs. These costs assume the hourly costs for required support equipment and personnel as discussed in Sections 16.10 and 16.11 respectively.

Table 21.15 Yearly Mine Support Costs

Mine Support Costs	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Consumables	K USD	$ 719	$ 2,293	$ 2,293	$ 2,299	$ 2,293	$ 2,293	$ 2,293	$ 2,299	$ 2,293	$ 2,293	$ 949	$ -	$ 22,315
Parts / MARC Cost	K USD	$ 255	$ 763	$ 763	$ 766	$ 763	$ 763	$ 763	$ 766	$ 763	$ 763	$ 316	$ -	$ 7,446
Maintenance Labor	K USD	$ 283	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 334	$ -	$ 7,839
Operating Labor	K USD	$ 566	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 669	$ -	$ 15,679
Total	K USD	$ 1,824	$ 5,463	$ 5,463	$ 5,472	$ 5,463	$ 5,463	$ 5,463	$ 5,472	$ 5,463	$ 5,463	$ 2,267	$ -	$ 53,279
Total After Capitalization	K USD	$ -	$ 5,463	$ 5,463	$ 5,472	$ 5,463	$ 5,463	$ 5,463	$ 5,472	$ 5,463	$ 5,463	$ 2,267	$ -	$ 51,455
Cost per Tonne Mined														
Consumables	$/t	$ 0.52	$ 0.13	$ 0.10	$ 0.08	$ 0.07	$ 0.09	$ 0.13	$ 0.14	$ 0.14	$ 0.15	$ 0.24	$ -	$ 0.11
Maintenance Allocations	$/t	$ 0.19	$ 0.04	$ 0.03	$ 0.03	$ 0.02	$ 0.03	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.08	$ -	$ 0.04
Maintenance Labor	$/t	$ 0.21	$ 0.05	$ 0.04	$ 0.03	$ 0.02	$ 0.03	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.09	$ -	$ 0.04
Operating Labor	$/t	$ 0.41	$ 0.09	$ 0.07	$ 0.06	$ 0.05	$ 0.07	$ 0.09	$ 0.09	$ 0.10	$ 0.11	$ 0.17	$ -	$ 0.08
Total Costs	$/t	$ 1.33	$ 0.31	$ 0.25	$ 0.19	$ 0.17	$ 0.22	$ 0.32	$ 0.32	$ 0.32	$ 0.37	$ 0.58	$ -	$ 0.27
Total After Capitalization	$/t	$ -	$ 0.31	$ 0.25	$ 0.19	$ 0.17	$ 0.22	$ 0.32	$ 0.32	$ 0.32	$ 0.37	$ 0.58	$ -	$ 0.26

21.6 Process Operating Cost Summary

Process operating costs were developed using 1) the installed equipment horse power at a electricity unit rate of $0.06 per kWh, 2) test work reagent consumptions and market pricing, 3) the proposed staffing plan and local labor rates, 4) estimated wear iron consumption, and 5) estimated rates of use and market pricing for consumables.

Power costs were estimated using the installed equipment motor power, less the power allocated for installed spares, to calculate the operating power load. Annual kWh were determined based on 100%

mechanical utilization and the annual power cost determined. A 5% escalation was used for equipment and loads not included in the process equipment list to estimate the total annual power cost for each processing circuit and the associated power unit cost per tonne. Summary power costs for the three phases are given in Table 21.16, Table 21.17, and Table 21.18.

Table 21.16 Florida Mountain Heap Leach - Power Costs

Item	Installed	Operating
Operating Power (kW)	6,666	5,700
Annual Operating Hours	8,760	8,760
Annual KWh	58,396,403	49,928,924
Power Costs $/KWh	0.06	0.06
Annual Power Cost	$3,503,784	$2,995,735
Equipment Allowance	5%	5%
Estimated Annual Power Cost	$3,678,973	$3,145,522
Power Costs $/t	$0.34	$0.29

Table 21.17 Florida Mountain Concentrator - Power Costs

Item	Installed	Operating
Operating Power (kW)	3,952	2,660
Annual Operating Hours	8,760	8,760
Annual KWh	34,615,683	23,303,145
Power Costs $/KWh	0.06	0.06
Annual Power Cost	$2,076,941	$1,398,189
Equipment Allowance	5%	5%
Estimated Annual Power Cost	$2,180,788	$1,468,098
Power Costs $/t	$3.03	$2.04

Table 21.18 DeLamar Heap Leach - Power Costs

Item	Installed	Operating
Operating Power (kW)	8,382	7,167
Annual Operating Hours	8,760	8,760
Annual KWh	73,426,811	62,779,923
Power Costs $/KWh	0.06	0.06
Annual Power Cost	$4,405,609	$3,766,795
Equipment Allowance	5%	5%
Estimated Annual Power Cost	$4,625,889	$3,955,135
Power Costs $/t	$0.43	$0.37

Reagent and consumable costs for the three processing phases are shown in Table 21.19, Table 21.20, and Table 21.21. Reagent costs are based on test work consumptions and InfoMine consumable cost indices.

Table 21.19 Florida Mountain Heap Leach Consumables Costs

Item	Reagent Cost	Cost Units	Source	Usage (kg/t)	Total Cost ($000/year)	Unit Cost ($/t)	% of Total
Crusher Liners	n/a	US$	Estimate; 12@$15000 for cones, 1@$130K for Gyro Concave, 2@70K for Mantles	n/a	450	0.04	2.90%
Cyanide	2.3	US$/kg	Market Price/Test Work Consumption	0.4	9,936	0.92	63.80%
Zinc Dust	4.38	US$/kg	Market Price/Test Work Consumption	0.01	473	0.04	3.00%
Lime	0.24	US$/kg	Market Price/Test Work Consumption	1	2,592	0.24	16.70%
Cement	0.35	US$/kg	Market Price/Test Work Consumption	0	-	0	0.00%
Anti-Scalent	2	US$/kg	Market Price/Test Work Consumption	0.01	216	0.02	1.40%
Refining Supplies		US$/kg	Allowance		400	0.04	2.60%
Maintenance Supplies		US$/kg	Allowance		1,000	0.09	6.40%
Laboratory Supplies		US$/kg	Allowance		500	0.05	3.20%
Total Reagents					**15,567**	**1.44**	**100.00%**

Table 21.20 Florida Mountain Concentrator with Concentrate Leach Consumables Costs

Item	Reagent Cost	Cost Units	Source	Usage (kg/t)	Total Cost ($000/year)	Unit Cost ($/t)	% of Total
Crusher Liners	n/a	US$	Estimate; 3@$5000 for cones, 1@$6000 for Jaw	n/a	21	0.03	1.50%
Ball Mill Balls	n/a	US$	Estimate based on Est. Abrasion Index	n/a	375	0.52	26.90%
Ball Mill Liners	n/a	US$	Estimate based on Est. Abrasion Index	n/a	35	0.05	2.50%
Collector A-208	2.4	US$/kg	Market Price/Test Work Consumption	0.05	86	0.12	6.20%
Collector PAX	1.61	US$/kg	Market Price/Test Work Consumption	0.03	29	0.04	2.10%
Aerofroth 65	1.8	US$/kg	Market Price/Test Work Consumption	0.1	130	0.18	9.30%
Lime	0.24	US$/kg	Market Price/Test Work Consumption	0.2	35	0.05	2.50%
Sodium Cyanide	2.3	US$/kg	Market Price/Test Work Consumption	0.2	331	0.46	23.80%
Flocculant total	2.09	US$/kg	Market Price/Test Work Consumption	0.05	75	0.1	5.40%
Maintenance Supplies		US$/kg	Allowance		200	0.28	14.40%
Laboratory Supplies		US$/kg	Allowance		75	0.1	5.40%
Total Reagents					**1,393**	**1.93**	**100.00%**

Table 21.21 DeLamar Heap Leach Consumables Costs

Item	Reagent Cost	Cost Units	Source	Usage (kg/t)	Total Cost ($000/year)	Unit Cost ($/t)	% of Total
Crusher Liners	n/a	US$	Estimate; 20@$15000 for cones, 1@$130K for Gyro Concave, 2@70K for Mantles	n/a	570	0.05	2.30%
Cyanide	2.3	US$/kg	Market Price/Test Work Consumption	0.4	9,936	0.92	40.70%
Zinc Dust	4.38	US$/kg	Market Price/Test Work Consumption	0.01	473	0.04	1.90%
Lime	0.24	US$/kg	Market Price/Test Work Consumption	0		0	0.00%
Cement	0.35	US$/kg	Market Price/Test Work Consumption	3	11,340	1.05	46.40%
Anti-Scalent	2	US$/kg	Market Price/Test Work Consumption	0.01	216	0.02	0.90%
Refining Supplies		US$/kg	Allowance		400	0.04	1.60%
Maintenance Supplies		US$/kg	Allowance		1,000	0.09	4.10%
Laboratory Supplies		US$/kg	Allowance		500	0.05	2.00%
Total Reagents					**24,435**	**2.26**	**100.00%**

Summary operating costs for the process development phases are shown in Table 21.22, Table 21.23 and Table 21.24.

Operations manpower for heap leach operation are $0.51 per tonne for Florida Mt, and $0.60 per tonne for DeLamar. The cost increase of $0.09 per tonne for DeLamar will be due to the addition of the tertiary crushing and agglomeration circuits and associated manpower requirements for operations and maintenance. Reagents costs increase from the Florida Mountain cost of $1.44 to $2.26 per tonne for DeLamar as a result of the use of cement for agglomeration. Power costs increase for DeLamar due to the increase in installed horsepower for the tertiary crushing and agglomeration circuits. The installed power increase, at roughly 2,300 hp, results in a power cost increase from $0.29 per tonne to $0.37 per tonne for Florida Mt. and DeLamar, respectively.

Table 21.22 Florida Mountain Heap Leach OPEX Summary

Item	Units	Cost	Percent of Total
Operations Manpower	$/t	0.51	22.8%
Reagents/Supplies	$/t	1.44	64.2%
Power	$/t	0.29	13.0%
TOTAL - Operating	**$/t**	**2.25**	**100%**

Table 21.23 Florida Mountain Concentrator OPEX Summary

Item	Units	Cost	Percent of Total
Operations Manpower	$/t	5.07	56.0%
Reagents/Supplies	$/t	1.94	21.5%
Power	$/t	2.04	22.5%
TOTAL - Operating	**$/t**	**9.05**	**100.0%**

Table 21.24 DeLamar Heap Leach OPEX Summary

Item	Units	Cost	Percent of Total
Operations Manpower	$/t	0.60	18.6%
Reagents/Supplies	$/t	2.26	70.1%
Power	$/t	0.37	11.3%
TOTAL - Operating	**$/t**	**3.23**	**100.0%**

Staffing levels for the Florida Mountain concentrator are estimated to be higher than would normally be expected for a concentrator rated for 2,000 tonnes per day due to the nature of the processing circuit. Reagents are also estimated to be higher owing to the cyanide required for leaching and flocculant associated with the CCD circuit.

MDA used the costs provided by Mr. Woods to determine yearly processing costs. Operations manpower costs in Table 21.22, Table 21.23, and Table 21.24 were applied as fixed costs based on 30 days per month, with daily throughput of 30,000 TPD for heap leaching and 2,000 TPD for the concentrator. Table 21.25 shows the yearly processing costs. Due to the application of the fixed costs and some periods that fell short of the targeted tonnage rates, the resulting cost per tonne differs slightly from those used to build up the costs.

Note that preproduction costs are capitalized as part of the owner's costs (see Section 21.3.2). The resulting LOM processing cost is $289.0 million. Also note that for the Florida Mountain heap leach material, the resulting cost is slightly less than that specified by Mr. Woods in Table 21.22 due to the capitalization of preproduction costs.

Table 21.25 Yearly Process Operating Costs

Florida Mountain Leach Cost	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Labor	K USD	$ 918	$ 5,508	$ 5,508	$ 5,508	$ 5,508	$ 2,295	$ -	$ -	$ -	$ -	$ -	$ -	$ 25,245
Reagents / Maintenance	K USD	$ 674	$ 11,941	$ 13,997	$ 14,035	$ 13,997	$ 10,142	$ 1,205	$ -	$ -	$ -	$ -	$ -	$ 65,991
Power	K USD	$ 136	$ 2,405	$ 2,819	$ 2,827	$ 2,819	$ 2,043	$ 243	$ -	$ -	$ -	$ -	$ -	$ 13,290
Total Fl Mnt Leach Costs	K USD	$ 1,728	$ 19,853	$ 22,324	$ 22,370	$ 22,324	$ 14,480	$ 1,448	$ -	$ -	$ -	$ -	$ -	$ 104,526
Total After Capitalized Costs	K USD	$ -	$ 19,853	$ 22,324	$ 22,370	$ 22,324	$ 14,480	$ 1,448	$ -	$ -	$ -	$ -	$ -	$ 102,798
Total After Capitalized Costs	$/t	$ -	$ 2.39	$ 2.30	$ 2.30	$ 2.30	$ 2.06	$ 1.73	$ -	$ -	$ -	$ -	$ -	$ 2.24
DeLamar Leach Costs														
Labor	K USD	$ -	$ -	$ -	$ -	$ -	$ 3,780	$ 6,480	$ 6,480	$ 6,480	$ 6,480	$ 2,700	$ -	$ 32,400
Reagents / Maintenance	K USD	$ -	$ -	$ -	$ -	$ -	$ 6,050	$ 20,043	$ 22,027	$ 21,967	$ 21,967	$ 6,520	$ -	$ 98,574
Power	K USD	$ -	$ -	$ -	$ -	$ -	$ 990	$ 3,281	$ 3,606	$ 3,596	$ 3,596	$ 1,067	$ -	$ 16,138
Total DeLamar Leach Costs	K USD	$ -	$ -	$ -	$ -	$ -	$ 10,820	$ 29,804	$ 32,114	$ 32,044	$ 32,044	$ 10,288	$ -	$ 147,113
Total After Capitalized Costs	K USD	$ -	$ -	$ -	$ -	$ -	$ 10,820	$ 29,804	$ 32,114	$ 32,044	$ 32,044	$ 10,288	$ -	$ 147,113
Total After Capitalized Costs	$/t	$ -	$ -	$ -	$ -	$ -	$ 4.04	$ 3.36	$ 3.29	$ 3.30	$ 3.30	$ 3.57	$ -	$ 3.37
Florida Mountain Mill Costs														
Labor	K USD	$ -	$ -	$ -	$ 3,042	$ 3,650	$ 3,650	$ 3,650	$ 3,650	$ 3,650	$ 608	$ -	$ -	$ 21,902
Reagents / Maintenance	K USD	$ -	$ -	$ -	$ 1,171	$ 1,397	$ 1,397	$ 1,397	$ 1,401	$ 1,397	$ 195	$ -	$ -	$ 8,354
Power	K USD	$ -	$ -	$ -	$ 1,231	$ 1,469	$ 1,469	$ 1,469	$ 1,473	$ 1,469	$ 205	$ -	$ -	$ 8,784
Total Florida Mnt Mill Costs	K USD	$ -	$ -	$ -	$ 5,444	$ 6,516	$ 6,516	$ 6,516	$ 6,524	$ 6,516	$ 1,008	$ -	$ -	$ 39,040
Total After Capitalized Costs	K USD	$ -	$ -	$ -	$ 5,444	$ 6,516	$ 6,516	$ 6,516	$ 6,524	$ 6,516	$ 1,008	$ -	$ -	$ 39,040
Total After Capitalized Costs	$/t	$ -	$ -	$ -	$ 9.02	$ 9.05	$ 9.05	$ 9.05	$ 9.04	$ 9.05	$ 10.04	$ -	$ -	$ 9.07
Total Project Processing Cost														
Labor	K USD	$ 918	$ 5,508	$ 5,508	$ 8,550	$ 9,158	$ 9,725	$ 10,130	$ 10,130	$ 10,130	$ 7,088	$ 2,700	$ -	$ 79,547
Reagents / Maintenance	K USD	$ 674	$ 11,941	$ 13,997	$ 15,206	$ 15,394	$ 17,589	$ 22,645	$ 23,428	$ 23,364	$ 22,162	$ 6,520	$ -	$ 172,919
Power	K USD	$ 136	$ 2,405	$ 2,819	$ 4,058	$ 4,288	$ 4,502	$ 4,993	$ 5,079	$ 5,065	$ 3,801	$ 1,067	$ -	$ 38,212
Total Project Costs	K USD	$ 1,728	$ 19,853	$ 22,324	$ 27,814	$ 28,840	$ 31,816	$ 37,768	$ 38,637	$ 38,560	$ 33,051	$ 10,288	$ -	$ 290,678
Total After Capitalized Costs	K USD	$ -	$ 19,853	$ 22,324	$ 27,814	$ 28,840	$ 31,816	$ 37,768	$ 38,637	$ 38,560	$ 33,051	$ 10,288	$ -	$ 288,951
Total After Capitalized Costs	$/t	$ -	$ 2.39	$ 2.30	$ 2.69	$ 2.76	$ 3.05	$ 3.62	$ 3.69	$ 3.69	$ 3.37	$ 3.57	$ -	$ 3.08

21.7 G&A Costs

G&A costs were estimated based on personnel requirements for administrative, accounting, safety and security, and environmental departments to support mining and processing activities. Costs are also included for legal, land, permit bonding, power, etc. Table 21.26 shows the yearly G&A cost estimate.

Table 21.26 Yearly G&A Costs

Personnel Costs	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Construction Management Personnel	K USD	$ 256	$ 26	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 281
Admin Salaried Personnel	K USD	$ 267	$ 504	$ 504	$ 504	$ 504	$ 504	$ 504	$ 504	$ 504	$ 504	$ 417	$ -	$ 5,217
Admin Hourly Personnel	K USD	$ 272	$ 580	$ 580	$ 580	$ 580	$ 580	$ 580	$ 580	$ 580	$ 580	$ 314	$ -	$ 5,802
Safety & Security Salaried Personnel	K USD	$ 45	$ 90	$ 90	$ 90	$ 90	$ 90	$ 90	$ 90	$ 90	$ 90	$ 82	$ -	$ 934
Safety & Security Hourly Personnel	K USD	$ 243	$ 649	$ 649	$ 649	$ 649	$ 649	$ 649	$ 649	$ 649	$ 649	$ 351	$ -	$ 6,434
Environmental Salaried Personnel	K USD	$ 92	$ 110	$ 110	$ 110	$ 110	$ 110	$ 110	$ 110	$ 110	$ 110	$ 101	$ -	$ 1,187
Environmental Hourly Personnel	K USD	$ 127	$ 152	$ 152	$ 152	$ 152	$ 152	$ 152	$ 152	$ 152	$ 152	$ 140	$ -	$ 1,637
Human Resources	K USD	$ 138	$ 207	$ 104	$ 104	$ 104	$ 104	$ 104	$ 104	$ 104	$ 104	$ 43	$ -	$ 1,216
Total Personnel Costs	K USD	$ 1,439	$ 2,317	$ 2,188	$ 2,188	$ 2,188	$ 2,188	$ 2,188	$ 2,188	$ 2,188	$ 2,188	$ 1,448	$ -	$ 22,708

General G&A Costs														
Construction Management Expenses	K USD	$ 200	$ 20	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 220
Supplies & General Maintenance	K USD	$ 80	$ 120	$ 120	$ 120	$ 120	$ 120	$ 120	$ 120	$ 120	$ 120	$ 110	$ -	$ 1,270
Land Holdings	K USD	$ 214	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 295	$ -	$ 3,404
Off Site Overhead	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Legal, Audits, Consulting, MSHA	K USD	$ 57	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 78	$ -	$ 900
Idaho State Property Tax	K USD	$ -	$ 485	$ 381	$ 563	$ 440	$ 329	$ 226	$ 124	$ 54	$ 18	$ -	$ -	$ 2,620
Computers, IT, Internet, Software, Hardware	K USD	$ 50	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 69	$ -	$ 794
Environmental, Montoring Wells, Reporting	K USD	$ 167	$ 250	$ 250	$ 250	$ 250	$ 250	$ 250	$ 250	$ 250	$ 250	$ 229	$ -	$ 2,646
Bond Carry Cost	K USD	$ 247	$ 245	$ 243	$ 242	$ 240	$ 238	$ 236	$ 234	$ 232	$ 230	$ 135	$ 134	$ 2,656
Donations, Dues, Public Relations	K USD	$ 20	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 28	$ -	$ 318
Insurance (Excluding Workmans Comp)	K USD	$ 1,217	$ 930	$ 930	$ 1,130	$ 1,130	$ 1,130	$ 1,130	$ 1,130	$ 1,130	$ 1,130	$ 1,130	$ -	$ 12,119
Travel, Lodging, Meals, Entertainment	K USD	$ 67	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 92	$ -	$ 1,058
Telephones, Computers, Cell Phones	K USD	$ 50	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 69	$ -	$ 794
Light Vehicle Maintenance, Fuel	K USD	$ 20	$ 43	$ 43	$ 43	$ 43	$ 43	$ 43	$ 43	$ 43	$ 43	$ 20	$ -	$ 426
Small Tools, Janitorial, Safety Supplies	K USD	$ 57	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 78	$ -	$ 900
Equipment Rentals	K USD	$ 67	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 92	$ -	$ 1,058
Access Road Maintenance	K USD	$ 100	$ 150	$ 150	$ 150	$ 150	$ 150	$ 150	$ 150	$ 150	$ 150	$ 138	$ -	$ 1,588
Office Power	K USD	$ 40	$ 60	$ 60	$ 60	$ 60	$ 60	$ 60	$ 60	$ 60	$ 60	$ 55	$ -	$ 635
Total General G&A Costs	K USD	$ 2,652	$ 3,176	$ 3,049	$ 3,429	$ 3,304	$ 3,191	$ 3,087	$ 2,982	$ 2,911	$ 2,873	$ 2,617	$ 134	$ 33,405
Total G&A	K USD	$ 4,092	$ 5,493	$ 5,237	$ 5,617	$ 5,492	$ 5,379	$ 5,275	$ 5,170	$ 5,099	$ 5,061	$ 4,065	$ 134	$ 56,113
Total After Capitalization	K USD	$ -	$ 5,493	$ 5,237	$ 5,617	$ 5,492	$ 5,379	$ 5,275	$ 5,170	$ 5,099	$ 5,061	$ 4,065	$ 134	$ 52,021

22.0 ECONOMIC ANALYSIS

MDA has prepared this PEA for the DeLamar mining project, which includes operations at both DeLamar and Florida Mountain. Note that a preliminary economic assessment is preliminary in nature and it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves, and there is no certainty that the preliminary assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

A summary of the PEA results is shown in Table 22.1. Some economic highlights include:

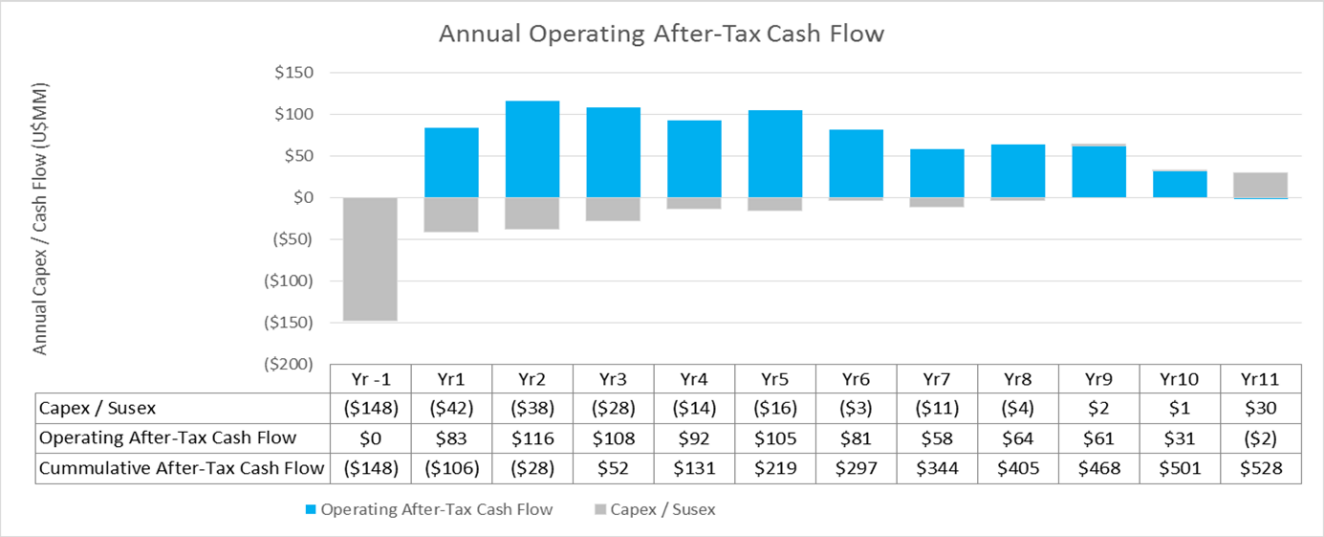
- Initial construction period is anticipated to be 18 months;
- After-tax net present value ("NPV") (5%) of $358 million with a 43% after-tax internal rate of return ("IRR") using $1,350 and $16.90 per ounce gold and silver prices, respectively;
- After-tax payback period of 2.35 years;
- Year 2 to 6 gold equivalent production of 148,000 ounces (126,000 oz Au and 1,796,000 oz Ag);
- Year 1 to 10 gold equivalent average production of 124,000 ounces (103,000 oz Au and 1,660,000 oz Ag).
- After-tax LOM cumulative cash flow of $528 million; and
- Average annual after-tax free cash flow of $61 million during production.

Table 22.1 summarizes the results of the economic analysis. Figure 22.1 shows the annual operating after-tax cash flow.

Table 22.1 Economic Analysis Summary

After-tax NPV (5%)	K USD	$357,572
After-tax NPV (8%)	K USD	$284,448
After-tax NPV (10%)	K USD	$244,454
After-tax IRR	%	43%
After-Tax Payback Period	Years	2.35

Figure 22.1 Annual Operating After-Tax Cash Flow



	Yr -1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11
Capex / Susex	($148)	($42)	($38)	($28)	($14)	($16)	($3)	($11)	($4)	$2	$1	$30
Operating After-Tax Cash Flow	$0	$83	$116	$108	$92	$105	$81	$58	$64	$61	$31	($2)
Cummulative After-Tax Cash Flow	($148)	($106)	($28)	$52	$131	$219	$297	$344	$405	$468	$501	$528

22.1 Mining Physicals

The cash-flow model uses the mining and process production schedule physicals summarized in Table 22.2. and as discussed in Section 16.9.

Table 22.2 Yearly Mine & Process Physicals

Material Mined	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Florida Mnt Above COG	K Tonnes	468	8,768	9,971	11,139	10,915	7,844	1,029	-	-	-	-	-	50,133
	g Au/t	0.36	0.50	0.48	0.45	0.46	0.49	0.42	-	-	-	-	-	0.47
	K Ozs Au	5	142	152	161	161	123	14	-	-	-	-	-	758
	g Ag/t	7.76	13.06	9.49	10.03	11.19	15.46	21.15	-	-	-	-	-	11.76
	K Ozs Ag	117	3,682	3,042	3,593	3,927	3,898	699	-	-	-	-	-	18,960
Florida Mnt Waste	K Tonnes	903	8,691	11,929	17,996	21,318	8,399	984	-	-	-	-	-	70,221
Florida Mnt Total Mined	K Tonnes	1,371	17,459	21,900	29,135	32,233	16,243	2,013	-	-	-	-	-	120,354
Florida Mnt Strip Ratio	W:O	1.93	0.99	1.20	1.62	1.95	1.07	0.96						1.40
DeLamar Above COG	K Tonnes	-	-	-	-	-	2,816	9,780	8,954	9,948	9,632	2,487	-	43,617
	g Au/t	-	-	-	-	-	0.35	0.36	0.33	0.35	0.35	0.35	-	0.35
	K Ozs Au	-	-	-	-	-	32	112	95	113	107	28	-	486
	g Ag/t	-	-	-	-	-	15.59	24.51	14.99	16.97	20.52	24.41	-	19.37
	K Ozs Ag	-	-	-	-	-	1,412	7,708	4,315	5,427	6,355	1,952	-	27,170
DeLamar Waste	K Tonnes	-	-	-	-	-	5,308	5,343	7,972	6,929	5,268	1,399	-	32,220
DeLamar Total Mined	K Tonnes	-	-	-	-	-	8,124	15,123	16,927	16,877	14,900	3,886	-	75,837
DeLamar Strip Ratio	W:O						1.88	0.55	0.89	0.70	0.55	0.56		0.74
Total Above COG	K Tonnes	468	8,768	9,971	11,139	10,915	10,660	10,809	8,954	9,948	9,632	2,487	-	93,750
	g Au/t	0.36	0.50	0.48	0.45	0.46	0.45	0.36	0.33	0.35	0.35	0.35	-	0.41
	K Ozs Au	5	142	152	161	161	155	126	95	113	107	28	-	1,244
	g Ag/t	7.76	13.06	9.49	10.03	11.19	15.49	24.19	14.99	16.97	20.52	24.41	-	15.30
	K Ozs Ag	117	3,682	3,042	3,593	3,927	5,310	8,408	4,315	5,427	6,355	1,952	-	46,130
Total Waste	K Tonnes	903	8,691	11,929	17,996	21,318	13,707	6,328	7,972	6,929	5,268	1,399	-	102,440
Total Mined	K Tonnes	1,371	17,459	21,900	29,135	32,233	24,367	17,136	16,927	16,877	14,900	3,886	-	196,190
Total Strip Ratio	W:O	1.93	0.99	1.20	1.62	1.95	1.29	0.59	0.89	0.70	0.55	0.56		1.09
Material Processed														
Total Leach	K Tonnes	468	8,292	9,720	9,747	9,720	9,720	9,706	9,747	9,720	9,720	2,885	-	89,444
	g Au/t	0.36	0.50	0.46	0.43	0.37	0.44	0.36	0.32	0.35	0.35	0.34	-	0.39
	K Ozs Au	5	133	143	135	116	136	113	101	111	108	31	-	1,132
	K Ozs Au Prod	-	97	130	115	101	119	90	78	84	86	36	-	936
	g Ag/t	7.76	13.31	9.41	9.86	9.49	15.37	24.78	15.07	16.99	20.33	23.40	-	15.21
	K Ozs Ag	117	3,549	2,941	3,091	2,966	4,802	7,733	4,721	5,309	6,352	2,170	-	43,751
	K Ozs Ag Prod	-	1,235	1,268	1,230	1,139	1,724	2,285	1,538	1,552	1,820	994	-	14,784
Florida Mountain Mill	K Tonnes	-	-	-	604	720	720	720	722	720	100	-	-	4,306
	g Au/t	-	-	-	1.27	1.38	0.69	0.60	0.51	0.51	0.51	-	-	0.80
	K Ozs Au	-	-	-	25	32	16	14	12	12	2	-	-	111
	K Ozs Au Prod	-	-	-	22	29	14	13	11	11	1	-	-	100
	g Ag/t	-	-	-	17.30	23.15	18.22	18.57	13.21	13.21	13.21	-	-	17.18
	K Ozs Ag	-	-	-	336	536	422	430	307	306	43	-	-	2,378
	K Ozs Ag Prod	-	-	-	269	429	337	344	245	245	34	-	-	1,903
Total Project	K Tonnes	468	8,292	9,720	10,350	10,440	10,440	10,426	10,469	10,440	9,820	2,885	-	93,750
	g Au/t	0.36	0.50	0.46	0.48	0.44	0.45	0.38	0.34	0.36	0.35	0.34	-	0.41
	K Ozs Au	5	133	143	159	148	152	127	113	122	110	31	-	1,244
	K Ozs Au Prod	-	97	130	137	130	133	102	89	95	87	36	-	1,036
	g Ag/t	7.76	13.31	9.41	10.30	10.43	15.56	24.35	14.94	16.73	20.25	23.40	-	15.30
	K Ozs Ag	117	3,549	2,941	3,426	3,502	5,224	8,163	5,028	5,615	6,394	2,170	-	46,130
	K Ozs Ag Prod	-	1,235	1,268	1,498	1,568	2,061	2,629	1,783	1,796	1,854	994	-	16,686

22.2 Pre-Tax Cash Flow

The pre-tax cash-flow model is shown in Table 22.3. This is based on the mining physicals shown in Table 22.2 along with the applications of metal prices discussed in Section 19.0 and the operating and capital costs discussed in Section 21.0 The revenues are based on $1,350 and $16.90 per ounce gold and silver prices, respectively. Transportation and refining costs are assumed to be $5.00 per ounce of gold and $0.50 per ounce of silver produced. Royalties have been applied as NSR royalties described in Section 4.3.

The total net revenue prior to application of costs is $1.64 billion. Subtracting the total operating costs of $733.2 million, $20.0 million in reclamation costs, and capital costs of $250.3 million yields a pre-tax LOM cash flow of $637.9 million (apparent discrepancies are due to rounding). The pre-tax LOM NPV at a 5.0% discount is estimated at $437.3 million, for a pre-tax internal rate of return ("IRR") of 49%.

Table 22.3 Pre-Tax Cash Flow

Revenues	**Units**	**Pre-Prod**	**Yr_1**	**Yr_2**	**Yr_3**	**Yr_4**	**Yr_5**	**Yr_6**	**Yr_7**	**Yr_8**	**Yr_9**	**Yr_10**	**Yr_11**	**Total**
Payable Metal - Au	K Ozs Au	-	96	130	136	129	133	102	88	94	87	36	-	1,031
Gold Revenue	K USD	$ -	$ 130,082	$ 175,289	$ 184,056	$ 174,717	$ 179,139	$ 137,225	$ 119,292	$ 127,018	$ 117,083	$ 48,189	$ -	$ 1,392,092
Transp. & Refining - Au	K USD	$ -	$ 482	$ 649	$ 682	$ 647	$ 663	$ 508	$ 442	$ 470	$ 434	$ 178	$ -	$ 5,156
Payable Metal - Ag	K Ozs Au	-	1,229	1,262	1,491	1,560	2,051	2,616	1,774	1,787	1,845	989	-	16,603
Silver Revenue	K USD	$ -	$ 20,772	$ 21,328	$ 25,193	$ 26,361	$ 34,663	$ 44,204	$ 29,980	$ 30,204	$ 31,174	$ 16,707	$ -	$ 280,585
Transp. & Refining - Ag	K USD	$ -	$ 615	$ 631	$ 745	$ 780	$ 1,026	$ 1,308	$ 887	$ 894	$ 922	$ 494	$ -	$ 8,301
Revenue Before Royalties	K USD	$ -	$ 149,757	$ 195,337	$ 207,822	$ 199,651	$ 212,113	$ 179,613	$ 147,944	$ 155,858	$ 146,901	$ 64,223	$ -	$ 1,659,220
Royalties	K USD	$ -	$ 460	$ 1,174	$ 1,195	$ 498	$ 2,277	$ 3,783	$ 3,352	$ 2,986	$ 1,450	$ 642	$ -	$ 17,816
Net Revenue	K USD	$ -	$ 149,297	$ 194,163	$ 206,628	$ 199,153	$ 209,836	$ 175,829	$ 144,592	$ 152,872	$ 145,451	$ 63,581	$ -	$ 1,641,404
Operating Costs														
Mining Costs	K USD	$ -	$ 37,661	$ 43,817	$ 53,197	$ 56,170	$ 47,180	$ 37,290	$ 35,039	$ 36,036	$ 34,314	$ 11,538	$ -	$ 392,243
Processing Costs														
Leaching	K USD	$ -	$ 19,853	$ 22,324	$ 22,370	$ 22,324	$ 25,300	$ 31,252	$ 32,114	$ 32,044	$ 32,044	$ 10,288	$ -	$ 249,911
Florida Mnt Milling	K USD	$ -	$ -	$ -	$ 5,444	$ 6,516	$ 6,516	$ 6,516	$ 6,524	$ 6,516	$ 1,008	$ -	$ -	$ 39,040
Total Processing Cost	K USD	$ -	$ 19,853	$ 22,324	$ 27,814	$ 28,840	$ 31,816	$ 37,768	$ 38,637	$ 38,560	$ 33,051	$ 10,288	$ -	$ 288,951
Other Costs														
G&A	K USD	$ -	$ 5,493	$ 5,237	$ 5,617	$ 5,492	$ 5,379	$ 5,275	$ 5,170	$ 5,099	$ 5,061	$ 4,065	$ 134	$ 52,021
Reclamation - Florida Mnt	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,750	$ 3,750	$ -	$ -	$ -	$ 7,500
Reclamation - DeLamar	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 12,500	$ 12,500
Total Reclamation	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,750	$ 3,750	$ -	$ -	$ 12,500	$ 20,000
Net Operating Cost	K USD	$ -	$ 63,007	$ 71,378	$ 86,628	$ 90,502	$ 84,375	$ 80,333	$ 82,597	$ 83,445	$ 72,426	$ 25,891	$ 12,634	$ 753,215
Net Operating Cash Flow	K USD	$ -	$ 86,291	$ 122,785	$ 120,000	$ 108,651	$ 125,461	$ 95,496	$ 61,995	$ 69,428	$ 73,025	$ 37,690	$ (12,634)	$ 888,188
Cumulative Op Cash Flow	K USD		$ 86,291	$ 209,076	$ 329,076	$ 437,727	$ 563,188	$ 658,685	$ 720,679	$ 790,107	$ 863,132	$ 900,822	$ 888,188	
Capital Costs														
Pre-Stripping	K USD	$ 7,514	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 7,514
Pre-Production Op Costs	K USD	$ 5,819	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 5,819
Mining Capital	K USD	$ 39,007	$ 27,950	$ 2,970	$ 13,629	$ -	$ 7,811	$ 400	$ -	$ -	$ -	$ -	$ -	$ 91,766
Process Capital	K USD	$ 62,579	$ 687	$ 34,887	$ 14,130	$ 13,764	$ 7,181	$ 2,890	$ 7,500	$ -	$ -	$ -	$ -	$ 143,618
Infrastructure / Owner's Capital	K USD	$ 27,070	$ -	$ -	$ -	$ -	$ 974	$ -	$ -	$ -	$ -	$ -	$ -	$ 28,044
Sub-Total	K USD	$ 141,989	$ 28,637	$ 37,856	$ 27,759	$ 13,764	$ 15,966	$ 3,290	$ 7,500	$ -	$ -	$ -	$ -	$ 276,761
Working Capital	K USD	$ -	$ 13,024	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (13,024)	$ -
Bonding Capital	K USD	$ 6,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (2,250)	$ -	$ (3,750)	$ -
Salvage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (1,088)	$ (25,337)	$ (26,426)
Total Capital	K USD	$ 147,989	$ 41,661	$ 37,856	$ 27,759	$ 13,764	$ 15,966	$ 3,290	$ 7,500	$ -	$ (2,250)	$ (1,088)	$ (42,111)	$ 250,336
Pre-Tax Cash Flow	K USD	$ (147,989)	$ 44,629	$ 84,929	$ 92,241	$ 94,887	$ 109,495	$ 92,206	$ 54,495	$ 69,428	$ 75,275	$ 38,778	$ 29,477	$ 637,852
Cumulative Pre-Tax Cash Flow	K USD	$ (147,989)	$ (103,359)	$ (18,430)	$ 73,811	$ 168,698	$ 278,193	$ 370,399	$ 424,894	$ 494,321	$ 569,596	$ 608,375	$ 637,852	
Pre-Tax Payback	Years		1.00	1.00	0.20	-	-	-	-	-	-	-	-	2.20

Pre-tax NPV (5%)	K USD	$437,310
Pre-tax NPV (8%)	K USD	$351,213
Pre-tax NPV (10%)	K USD	$304,072
Pre-tax IRR	%	49%

Note that reclamation costs of $20 million have been included in the above table as operating costs.

22.3 Tax Considerations & After-Tax Cash Flow

MDA and the author are not experts in taxation and have relied on Integra to provide tax treatment methodologies. Depreciation and depletion, along with the deduction of tax pools spent by Integra, have been applied to reduce the taxable income for the project.

Adjustments to the pretax LOM cash flow of $637.9 million were made as follows:

- 2019 exploration -$8.0 million;
- Existing tax credits -$22.7 million; and
- Estimated 2019/2020 G&A tax credits -$2.0 million.

This leaves an estimated book income of $605.1 million after tax credits. Depreciation and depletion further reduces the taxable income to $405.8 million. Estimated federal taxes (21%), Idaho State Tax (6.9%), and Idaho Mining Tax (1%) total $109.7 million. After deductions and credits, the resulting effective tax rate is 18.1%.

Tax calculations and the after-tax cash flow are shown in Table 22.4.

Table 22.4 Depreciation, Depletion, Taxes, and After-Tax Cash Flow

	Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Total
Depreciation	K USD	$ -	$ (146,838)	$ (31,096)	$ (19,293)	$ (9,652)	$ (8,737)	$ (6,325)	$ (6,652)	$ (6,793)	$ (5,931)	$ (4,579)	$ (3,393)	$ (249,289)
Amortization of Permitting	K USD	$ -	$ (1,000)	$ (1,000)	$ (1,000)	$ (1,000)	$ (1,000)	$ (1,000)	$ (1,000)	$ (1,000)	$ (1,000)	$ (1,000)	$ (1,000)	$ (11,000)
Amortization of Exploration	K USD	$ (1,352)	$ (1,352)	$ (1,352)	$ (1,352)	$ (1,352)	$ (1,334)	$ (800)	$ -	$ -	$ -	$ -	$ -	$ (8,892)
Amortization of Development	K USD	$ -	$ (793)	$ (1,033)	$ (1,099)	$ (1,056)	$ (1,123)	$ (953)	$ (784)	$ (826)	$ (779)	$ (341)	$ -	$ (8,788)
Equipment Salvage Gain												$ 1,088	$ 24,877	$ 25,966
Net Income before Depletion	K USD	$ (1,352)	$ (63,691)	$ 88,304	$ 97,256	$ 95,591	$ 113,266	$ 86,419	$ 53,559	$ 60,808	$ 65,315	$ 32,859	$ 7,851	$ 636,185
Depletion Allowed	K USD	$ -	$ 1,113	$ 28,682	$ 30,523	$ 29,421	$ 30,994	$ 25,966	$ 21,353	$ 22,577	$ 21,484	$ 9,391	$ -	$ 221,505
Net Operating Losses Deduction	K USD	$ -	$ -	$ (47,698)	$ (27,313)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (75,011)
Net Taxable Income	K USD	$ -	$ -	$ 11,924	$ 39,419	$ 66,171	$ 82,272	$ 60,452	$ 32,206	$ 38,231	$ 43,831	$ 23,468	$ 7,851	$ 405,825
Federal Taxes (21%)	K USD	$ -	$ -	$ 2,306	$ 7,622	$ 12,795	$ 15,908	$ 11,689	$ 6,227	$ 7,392	$ 8,475	$ 4,538	$ 1,518	$ 78,469
Idaho State Tax (6.9%)	K USD	$ -	$ 2,435	$ 4,056	$ 3,722	$ 3,141	$ 4,146	$ 2,443	$ 993	$ 1,558	$ 2,692	$ 1,481	$ 493	$ 27,158
Idaho Mining Tax (1%)	K USD	$ -	$ 454	$ 586	$ 537	$ 454	$ 599	$ 353	$ 143	$ 225	$ 389	$ 214	$ 71	$ 4,024
Total Tax	K USD	$ -	$ 2,888	$ 6,948	$ 11,881	$ 16,389	$ 20,653	$ 14,485	$ 7,363	$ 9,175	$ 11,555	$ 6,232	$ 2,082	$ 109,652

Pre-tax Cash Flow	K USD	$ (147,989)	$ 44,629	$ 84,929	$ 92,241	$ 94,887	$ 109,495	$ 92,206	$ 54,495	$ 69,428	$ 75,275	$ 38,778	$ 29,477	$ 637,852
Total Tax	K USD	$ -	$ 2,888	$ 6,948	$ 11,881	$ 16,389	$ 20,653	$ 14,485	$ 7,363	$ 9,175	$ 11,555	$ 6,232	$ 2,082	$ 109,652
Net After Tax Cash Flow	K USD	$ (147,989)	$ 41,741	$ 77,981	$ 80,360	$ 78,498	$ 88,842	$ 77,722	$ 47,131	$ 60,253	$ 63,720	$ 32,546	$ 27,395	$ 528,201
Cumulative After Tax Cash Flow	K USD	$ (147,989)	$ (106,247)	$ (28,266)	$ 52,094	$ 130,592	$ 219,434	$ 297,156	$ 344,287	$ 404,540	$ 468,259	$ 500,806	$ 528,201	

After-tax NPV (5%)	K USD	$357,572
After-tax NPV (8%)	K USD	$284,448
After-tax NPV (10%)	K USD	$244,454
After-tax IRR	%	43%
After-Tax Payback Period	Years	2.35

22.4 Sensitivity Analyses

Economic sensitivities of the project to changes in metal prices were evaluated based on constant gold to silver ratios as shown in Table 22.5. Sensitivity to revenue, operating costs, and capital costs were evaluated and are shown in Table 22.6, Table 22.7, and Table 22.8, respectively, showing the after-tax values. The after-tax sensitivity is shown in Figure 22.2. Revenue sensitivity assumes that the gold to silver price ratio remains the same.

A sensitivity analysis was completed for both gold and silver recoveries determine the impact on the economics with changes to recoveries. These results are shown in Table 22.9 and Table 22.10 for gold and silver, respectively. The change in recovery was applied to each of the processes (ROM for DeLamar and Florida Mountain and the Florida Mountain mill). For each 1% change in gold recovery there is a corresponding change in NPV (5%) of about $9.6 million and about 1% change to the IRR (decrease for decreasing recoveries and increase for increasing recoveries).

The change due to silver recover is about $4.1 million or 0.3% IRR for each 1% change in silver recovery.

Table 22.5 Project Sensitivity to Metal Prices

$/oz Au	$/oz Ag	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
$ 1,150	$ 14.40	$214,366	$161,805	$133,264	29%	3.28
$ 1,200	$ 15.02	$250,620	$192,878	$161,450	32%	2.98
$ 1,250	$ 15.65	$286,448	$223,583	$189,302	36%	2.72
$ 1,275	$ 15.96	$304,260	$238,842	$203,139	38%	2.61
$ 1,300	$ 16.27	$322,031	$254,044	$216,910	40%	2.52
$ 1,350	$ 16.90	$357,572	$284,448	$244,454	43%	2.35
$ 1,400	$ 17.53	$393,015	$314,775	$271,932	47%	2.20
$ 1,450	$ 18.15	$428,454	$345,098	$299,408	50%	2.06
$ 1,500	$ 18.78	$463,860	$375,375	$326,829	54%	1.94
$ 1,550	$ 19.40	$499,183	$405,553	$354,144	57%	1.84
$ 1,600	$ 20.03	$534,436	$435,676	$381,410	60%	1.76

Table 22.6 Revenue Sensitivity (After Tax)

% Change	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
70%	$59,753	$28,734	$12,268	12%	5.43
80%	$167,878	$121,792	$96,867	24%	3.81
90%	$265,945	$205,960	$173,283	34%	2.88
100%	$357,572	$284,448	$244,454	43%	2.35
110%	$448,026	$361,884	$314,642	52%	1.98
120%	$537,881	$438,706	$384,206	61%	1.74
130%	$627,737	$515,528	$453,771	70%	1.56

Table 22.7 Operating Cost Sensitivity (After Tax)

% Change	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
70%	$479,956	$388,299	$338,063	54%	1.93
80%	$439,335	$353,849	$307,024	51%	2.05
90%	$398,489	$319,177	$275,763	47%	2.19
100%	$357,572	$284,448	$244,454	43%	2.35
110%	$316,447	$249,557	$213,007	39%	2.53
120%	$274,539	$213,980	$180,930	35%	2.75
130%	$231,506	$177,465	$148,013	31%	3.03

Table 22.8 Capital Cost Sensitivity (After Tax)

% Change	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
70%	$429,247	$353,558	$311,788	70%	1.55
80%	$405,355	$330,521	$289,343	59%	1.80
90%	$381,464	$307,485	$266,898	50%	2.07
100%	$357,572	$284,448	$244,454	43%	2.35
110%	$333,681	$261,411	$222,009	38%	2.66
120%	$309,789	$238,374	$199,564	33%	2.99
130%	$285,898	$215,337	$177,120	29%	3.33

Figure 22.2 After-Tax Sensitivity

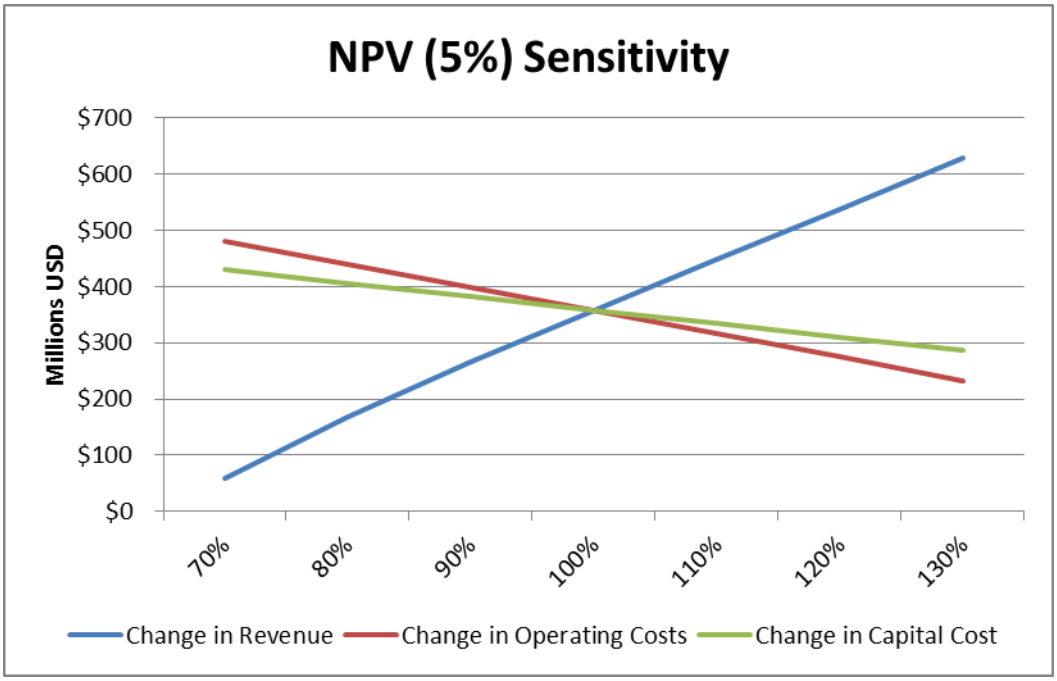


Table 22.9 Gold Recovery Sensitivity

Recovery Change	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
-5%	$309,790	$243,558	$207,400	38%	2.59
-4%	$319,347	$251,736	$214,811	39%	2.54
-3%	$328,903	$259,914	$222,222	40%	2.49
-2%	$338,459	$268,092	$229,632	41%	2.44
-1%	$348,025	$276,278	$237,051	42%	2.40
0%	$357,572	$284,448	$244,454	43%	2.35
1%	$367,104	$292,607	$251,848	44%	2.31
2%	$376,632	$300,763	$259,240	45%	2.27
3%	$386,385	$309,111	$266,807	46%	2.22
4%	$395,689	$317,075	$274,025	47%	2.18
5%	$405,216	$325,231	$281,416	48%	2.14

Table 22.10 Silver Recovery Sensitivity

Recovery Change	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
-5%	$336,945	$267,257	$229,139	42%	2.42
-4%	$341,070	$270,695	$232,202	42%	2.40
-3%	$345,196	$274,134	$235,265	42%	2.39
-2%	$349,321	$277,572	$238,328	43%	2.38
-1%	$353,447	$281,010	$241,391	43%	2.36
0%	$357,572	$284,448	$244,454	43%	2.35
1%	$361,686	$287,877	$247,509	44%	2.34
2%	$365,796	$291,303	$250,562	44%	2.33
3%	$369,906	$294,729	$253,614	44%	2.31
4%	$374,015	$298,154	$256,667	45%	2.30
5%	$378,125	$301,580	$259,719	45%	2.29

23.0 ADJACENT PROPERTIES

The authors have no information to report from adjacent properties.

23.0 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any other relevant data and information needed to make this report not misleading.

24.0 INTERPRETATION AND CONCLUSIONS

The authors have reviewed the data from the DeLamar project, which includes the DeLamar and Florida Mountain areas, and undertook verification of the data that are material to this report. Based on the work completed or supervised by the authors, it is the opinion of the authors that the project data are of sufficient quality for the modeling, estimation, and classification of the gold and silver resources disclosed in this report, as well as for the completion of the PEA summarized herein. Furthermore, the authors are unaware of any significant risks or uncertainties that could reasonably be expected to affect the reliability of the current mineral resources.

From 1891 through 1998, total production of gold and silver from the DeLamar – Florida Mountain project area is estimated to be approximately 1.3 million ounces of gold and 70 million ounces of silver. This includes an estimated 1.025 million ounces of gold produced from the original De Lamar underground mine and the later DeLamar open-pit operation. At Florida Mountain, nearly 260,000 ounces of gold and 18 million ounces of silver were produced from the historical underground mines and late 1990s open-pit mining.

The first precious-metals production of significance from the DeLamar project occurred from underground mines that exploited high-grade veins in the late 1800s to early 1900s. A total of 553,000 ounces of gold and 21.3 million ounces of silver were reportedly produced from the De Lamar and Florida Mountain underground mines during this time period. Beginning in the late 1970s, lower-grade bulk-tonnage ores were produced from an open-pit mining and milling operation that operated over a period of 21 years. Including the Florida Mountain operation that commenced in 1994, the combined DeLamar and Florida Mountain production from 1977 through 1998 was approximately 750,000 ounces of gold and 47.6 million ounces of silver. Extensive reclamation of the mining areas has been undertaken prior to Integra's involvement in the project, and Integra will continue related water-management activities, monitoring, and reporting to the appropriate governmental agencies.

The DeLamar project gold and silver deposits are characterized as volcanic-hosted, low-sulfidation epithermal mineralization. Higher-grade, steeply dipping vein-type mineralization is structurally controlled and therefore relatively restricted in widths, although some of the principal mineralized structures in the DeLamar area persist for 100s of meters along strike. At Florida Mountain, essentially the entire deposit is comprised of relatively thin, sub-vertical, mineralized structures of all grade ranges that form broad zones with significant strike extents, although individual higher-grade structures within these zones typically have limited strike lengths. At the DeLamar area, low-angle higher-grade zones commonly extend outwards for significant distances from the steeply dipping mineralization along and/or subparallel to certain contacts in the felsic volcanic stratigraphy. In the central portion of the DeLamar area that was the focus of all historical mining, modeled gold and silver resources extend continuously for approximately three kilometers of strike length, a maximum northeast-southwest width of 1.2 kilometers, and an elevation range of 570 meters. The Milestone area of the DeLamar mineralization adds an additional 640 meters of strike to the resource modeling. Resources at Florida Mountain have a northerly strike extent of about 1.3 kilometers, an east-west width of up to 650 meters, and an elevation range of 465 meters.

The project databases include the data from 2,625 generally shallow, historical conventional rotary, RC, and core holes drilled between 1966 and 1998 by various operators, for a total of 275,790 meters of

drilling. These holes have an average down-hole depth of less than 100 meters in the DeLamar area and 130 meters at Florida Mountain. Integra added 93 RC and core holes for a total of 30,288 meters to the project databases.

Results from the ongoing metallurgical testing program show that oxide and transitional materials from both the DeLamar and Florida Mountain deposits can be processed by heap-leach cyanidation, with no need for agglomeration pretreatment of Florida Mountain material, where production starts, but possibly needed for portions of the DeLamar area mineralization. Unoxidized material from the Florida Mountain deposit is amenable to grinding followed by agitated cyanide leaching, as well as to upgrading by gravity and subsequent flotation of the gravity tails. Certain portions of DeLamar unoxidized material show good responses to grinding and agitated leach, whereas other portions of DeLamar unoxidized material has shown highly variable responses to grinding followed by agitated cyanidation, with generally low gold and silver recoveries. Additional testing and mineralogic studies are needed to gain a better understanding of the observed variability in recoveries within the DeLamar unoxidized material. DeLamar unoxidized material generally responds well to upgrading by gravity and flotation processing. Testing to evaluate subsequent processing of the resulting concentrate is in progress, but results are not available as of the effective date of this report.

Potential open-pit gold and silver resources at the DeLamar project are constrained to lie within optimized pits and are tabulated using a cutoff grade of 0.2 g AuEq/t for oxidized and transitional materials and 0.3 g AuEq/t for unoxidized mineralization. Parameters used in the pit optimizations and cutoff grades reflect potential heap leaching of the oxidized and transitional mineralized materials, with parameters for unoxidized mineralization reflecting potential processing by agitated tank leaching with cyanide at Florida Mountain and by flotation concentration and off-site treatment at DeLamar. Project-wide Measured and Indicated resources total 172,365,000 tonnes averaging 0.43 g Au/t (2,376,000 ounces of gold) and 21.0 g Ag/t (116,514,000 ounces of silver). Inferred resources total 28,266,000 tonnes at an average grade of 0.38 g Au/t (343,000 ounces of gold) and 13.5 g Ag/t (12,240,000 ounces of silver).

The classification of the project resources has been upgraded significantly from the prior estimates. This reflects enhanced geological inputs into the resource modeling, importantly including detailed oxidation modeling, an increase in the understanding of the historical data, the addition of Integra's drill data, and a more sophisticated approach to the resource modeling.

Lower grade oxide and transition material from DeLamar and Florida Mountain is amenable for processing by crushed-ore cyanide heap leaching. Gold and silver leached at a central heap-leach facility will be extracted by Merrill-Crowe zinc precipitation.

For the higher-grade Florida Mountain material, processing will involve grinding followed by gravity and flotation concentration, with the concentrates processed by regrinding, agitated leaching, counter CCD, and Merrill-Crowe zinc precipitation of gold and silver. Flotation tailings will be dry stacked at a central tailings storage facility. Concentrate leach tailings will be added to the heap-leach circuit for the recovery precious metals.

The PEA presented in this report considers open-pit mining of the DeLamar and Florida Mountain gold-silver deposits using conventional truck and shovel operations over a 10-year mine life that follows an

initial 18 months of preproduction work. Process rates used in the PEA were 27,000 tonnes per day, or 9,720,000 tonnes per year, for cyanide heap leaching and 2,000 tonnes per day, or 720,000 tonnes per year, for flotation milling of the Florida Mountain unoxidized material. Leaching of Florida Mountain material would involve crushing, whereas DeLamar material would involve crushing and some agglomeration.

A gold price of $1,250 per ounce gold price was used to determine the ultimate pit limits for design, while a gold price of $1,350 per ounce was used for the PEA economic evaluation. The PEA total LOM gold production is estimated to be 1,036,000 ounces, with LOM average recovery of 83%. Silver production is estimated to be 16,686,000 ounces, with an average LOM recovery of 36%. The DeLamar area pits have a total of 32.2 million tonnes of waste associated with the material to be processed, and thus have an overall strip ratio of 0.74 tonnes of waste per tonne processed. The Florida Mountain pit is associated with a total of 70.2 million tonnes of waste, resulting in a stripping ratio of 1.40 waste tonnes to processed tonnes.

Economic highlights of the PEA include (i) after-tax net present value ("NPV") (5%) of $358 million with a 43% after-tax internal rate of return ("IRR") using $1,350 and $16.90 per ounce gold and silver prices, respectively; (ii) after-tax payback period of 2.35 years; and (iii) year 1 to 10 average production of 124,000 oz AuEq (103,000 oz Au and 1,660,000 oz Ag) with year 2 to 6 production of 148,000 oz AuEq (126,000 oz Au and 1,796,000 oz Ag). The total cash cost is estimated to be $619 per ounce of gold equivalent and all-in sustaining costs are estimated to be $742 per ounce of gold equivalent. Note that a PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves. There is no certainty that the economic results of the PEA will be realized.

24.1 DeLamar Project Opportunities

Exploration potential for additional bulk-tonnage mineralization at the DeLamar project remains significant. Most of the modeled mineralization at both the DeLamar and Florida Mountain areas is open at depth and, in several areas, along strike as well, which creates the opportunity to expand the presently defined resources that are potentially minable by open-pit methods. For example, the main portion of the Florida Mountain resources encompass the Trade Dollar – Black Jack vein system that was the focus of historical underground mining. Historical stopes along this vein system extend for 600 meters beyond the southern limits of the resources, but few holes have tested this extension. The DeLamar area resources are open along strike to the south, and Integra has recently undertaken drilling in this area. While significant mineralization has been encountered, at present continuity has not been established and the geology is not fully understood, but further work is needed. The DeLamar area resources also remain open downdip to the southwest in much of the Glen Silver area, where the resource limits are defined by the lack of drill data.

In addition to the bulk-tonnage potential, the potential for the discovery of high-grade vein-type mineralization similar to that mined in the late 19th and early 20th centuries also remains. In the DeLamar area, historical underground and open-pit mining exploited high-grade veins in the Sommercamp and North DeLamar zones that include less than 500 meters of the total three kilometers of strike length of continuous mineralization. The Milestone area adds another 0.6 kilometers of near-

surface mineralization that lacks testing for deeper high-grade zones. At Florida Mountain, historical underground mining focused on the Trade Dollar-Black Jack vein system, which includes the Alpine vein. Historical records in the possession of Integra indicate that these veins were mined over a strike length of 1,800 meters and vertical extents up to 450 meters. The Florida Mountain mineral resources reported herein encompass only the uppermost, lower-grade portions of the Florida Mountain gold-silver vein systems, and do not include any contribution from deeper high-grade veins. In addition to the lower-grade mineralization associated with the upper elevations of the Trade Dollar-Black Jack vein system, the resources include similar mineralization along multiple other vein systems that lie west of the Trade Dollar-Black Jack veins, including the Ontario, Arcuate, Tip Top, and Stone Cabin vein structures. The style of mineralization in the higher-elevation gold-silver resources along these other mineralized zones is indistinguishable from that modeled along the Trade Dollar-Black Jack veins, but significant historical underground mining is not known to have occurred along these vein zones. The vein systems lying to the west of the Trade Dollar-Black Jack vein system therefore remain as exploration targets for high-grade, potentially underground-mineable mineralization.

There are occurrences of epithermal-style mineralization and/or alteration peripheral to the current resources that represent other exploration targets. The best example of these is the Town Road – Henrietta area southwest of the northwestern portion of the Glen Silver area. Significant epithermal alteration in outcrop, float, and in the Henrietta mine dump has been identified, and sparse drilling has encountered gold and silver mineralization. As the geologic understanding of this and other peripheral occurrences is enhanced, quality drill targets will likely be generated.

Opportunities to improve process recoveries and/or decrease process costs through continued metallurgical testing include:

- Evaluate the coarsening of crush sizes for heap leaching of the DeLamar and Florida Mountain oxide and transitional heap-leach feeds. Evaluation of two-stage crush, single-stage crush and ROM leaching will be considered;
- Evaluate the elimination of agglomeration pretreatment for certain DeLamar heap-leach material;
- Further optimization of the processing conditions for the planned heap leach and mill processes may improve recoveries or decrease reagent consumptions; and
- Evaluation of higher-grade oxide and transitional material types (particularly higher silver grade) for processing by grind-leach and flotation with concentrate regrind and leach to determine if any of these materials are better processed by milling.

The opportunity to add value to the project through the processing of the DeLamar unoxidized materials will include evaluation of the following:

- Identification and modeling of materials that can be processed by grind-leach (cyanidation), including optimization of grind-leach procedures and evaluating selective mining of the materials that can be so processed;
- Continued evaluation and optimization of flotation treatment for the material types not amenable to grind-leach;
- Evaluation and optimization of flotation concentrate processing; and

- Evaluation of off-site shipment of flotation concentrate for toll processing.

As with all gold and silver projects, an increase in metal prices will reflect an upside in total revenues for the project.

24.2 DeLamar Project Risks

Risks related to the metallurgy/mineral processing include:

- Variability testing may show that some material types have lower recoveries or higher reagent consumptions than indicated by the materials tested to date;
- Variability testing may reveal a need to reclassify some materials to different oxidation categories, which can adversely affect expected recoveries; and
- Problems related to high clay content in certain DeLamar material types may be severe enough to adversely affect projected recoveries.

No pit-slope geotechnical studies have been completed. Pit designs have used 45° slopes, and while previously mined pits are standing with slopes about the same overall angle, deeper pits may require some flattening of pit walls, which could increase required stripping of waste.

The mine site is located at fairly high altitudes that can receive substantial snowfall during some years. While some snow removal equipment has been included in the study, additional operating costs and potential shutdowns may occur during mining operations.

Finally, as with most precious-metal mining projects, there are risks to the project after-tax payback period, NPV, and IRR if gold and silver prices decrease during the LOM, as shown in the sensitivity analyses. Higher than expected capital and operating costs are additional risks to the project economics which could result in decreases to the after tax NPV and IRR.

25.0 RECOMMENDATIONS

As discussed in Section 24.0, there continues to be excellent potential to expand the extents of mineralization of economic interest within the DeLamar project, and the project therefore warrants significant additional investment. Drilling should be a significant component of future expenditures, including infill drilling, to obtain samples for the ongoing metallurgical program, and step-out drilling, focused on both expanding the existing limits of the current project resources and testing targets peripheral to the resources. By end of 2020, Integra anticipates completing a drill program comprising approximately 13,000 meters of exploration, expansion and infill RC and core drilling, and 7,500 meters of metallurgical / resource confirmation core drilling (Table 25.1).

This proposed work would necessitate a modification to the existing Notification for drilling in the DeLamar area, as well as a new Notification for Florida Mountain drilling performed on patented claims. A Notice would need to be filed with the BLM if any of the recommended drilling is undertaken on unpatented claims not covered by an existing Notice. Separate Notices would be filed with the BLM for each of the DeLamar and Florida Mountain areas of unpatented claims.

As a complement to the RC drilling, an exploration program should be conducted that includes soil sampling, geological mapping, and an IP survey(s) to assist in developing targets. The results of the IP survey completed at the DeLamar area proved its usefulness in identifying mineralized trends.

The ongoing metallurgical testing should be continued, with a focus on defining the metallurgical and mineralogical characteristics and variability for the oxide, transitional, and unoxidized materials for each of the DeLamar and Florida Mountain resources. Evaluation of processing alternatives should continue to be examined for the DeLamar unoxidized materials, and optimization of processing scenarios should be continued for each of the major material types for the DeLamar and Florida Mountain resources.

The continued collection of specific-gravity data from the proposed core drilling programs is highly recommended.

Geotechnical investigations need to be conducted for pit-slope stability, the heap-leach pad, tailings impoundment, crushing plant site alternatives, waste rock disposal sites, and borrow areas for clay. Following these investigations and analyses of results, preliminary design layouts should be advanced from conceptual designs to refine facility locations and construction estimates.

Estimated costs for the recommended work program outlined above are presented in Table 25.1. This program is for work to be completed through to the end of 2020, and it has an estimated total cost of $14,595,000, including work towards the completion of a preliminary feasibility study ("PFS") in 2021. The estimated drilling costs are all-inclusive, as they include Integra's labor costs, access and drill-pad construction costs, assaying, etc., in addition to the contractor costs. In addition to the technical programs, the costs include land holding fees, environmental permitting costs, project-site general and administrative costs, operation of the water-treatment plant, and ongoing site reclamation activities through December 2020.

Table 25.1 Integra Cost Estimate for the Recommended Program

Item	Estimated Cost US$
Exploration RC Drilling (6,500 meters)	$1,500,000
Infill RC and Core Drilling (6,500 meters)	$2,500,000
Metallurgical / Infill Core Drilling (7,500 meters)	$2,700,000
Geological Mapping, Soil Sampling, Geophysics	$375,000
Land Holding Costs	$320,000
Metallurgy	$1,000,000
Geotechnical Studies	$400,000
Resource Update, Initiate PFS, and Technical Report	$900,000
Permitting	$1,900,000
Care and Maintenance / Reclamation	$2,000,000
Site Maintenance and G&A	$1,000,000
Total	**$14,595,000**

It is the authors' opinion that the DeLamar project is a project of merit that warrants the proposed program and level of expenditures outlined above.

26.0 REFERENCES

Ahlrichs, J.W., 1978 (June), *Evaluation of Silver Losses in Leached Tailings and Examination of High-Grade Ore and Feed from the DeLamar Mine, Jordan Valley, Oregon*: report prepared by Newmont Exploration Limited Metallurgical Department Danbury, CT, File No. 200-01.

Armstrong, R.L., 1975, *The geochronometry of Idaho*: Isochron/West, no.14, 50 p.

Aseto, C.O., 2012, *Geology, geochemistry, and geochronology of low-sulfidation epithermal Au-Ag ores on War Eagle Mountain, Silver City District, Idaho*: unpublished M.S. thesis, Auburn University, 167 p.

Asher, R.R., 1968, Geology and mineral resources of a portion of the Silver City region, Owyhee County, Idaho: Idaho Bureau of Mines and Geology Pamphlet 138, 106 p.

Barrett, R.A., 1985, The geology, mineralization, and geochemistry of the Milestone hot–spring silver-gold deposit near the Delamar silver-gold mine, Owyhee County, Idaho: unpublished M.Sc. thesis, University of Idaho, 474 p.

Bennett, E.H., and Galbraith, J., 1975, Reconnaissance Geology and Geochemistry of the Silver City-South Mountain Region, Owyhee County, Idaho: Idaho Bureau of Mines and Geology Pamphlet 162, 88 p.

Bergendahl, M.H., 1964, Gold, in Mineral and Water Resources of Idaho: Idaho Bureau of Mines and Geology Special Report No. 1, p. 93-101.

Bonnichsen, B., 1983, Epithermal gold and silver deposits Silver City-De Lamar district, Idaho: Idaho Geological Survey Technical Report 83-4, 29 p.

Bonnichsen, B. and Godchaux, M.M., 2006, Geologic Map of the Murphy 30 x 60 degree Quadrangle, Ada, Canyon, Elmore, and Owyhee Counties, Idaho: Idaho Geological Survey DWM-80.

Bonnichsen, B., Strowd, W.B., and Beebe, M., undated; Epithermal gold and silver deposits, Silver City-De Lamar District, Idaho: unpublished report, 28 p.

Cupp, B.L., 1989, Mineralization and volcanism at the DeLamar Silver Mine, Owyhee County, Idaho: Unpublished M.Sc. thesis, Miami University, 95 p.

DeLong, R., 2017 (September), untitled project communication document containing text for Section 4.4, received via email on September 25, 2017.

DeLong, R., 2019 (July), untitled project communication document containing text for Section 4.4, received via email on July 17, 2019.

Earth Resources Company, 1974, Feasibility study, DeLamar project, Owyhee County, Idaho, Volume II, geology and ore reserves: unpublished report, 41 p. plus figures and folded plates.

Ekren, E.B., McIntyre, D.H., Bennett, E.H., and Malde, H.E., 1981, *Geologic map of Owyhee County, Idaho, west of longitude 116°W*: U.S. Geological Survey Map1-1256.

Ekren, E.B., McIntyre, D.H., Bennett, E.H., and Marvin, R.F., 1982, *Cenozoic stratigraphy of western Owyhee County, Idaho,* in Bonnichsen, B. and Breckenridge, R.M., eds., Cenozoic Geology of Idaho, Idaho Bureau of Mines and Geology Bulletin 26.

Ekren, E.B., McIntyre, D.H., and Bennett, E.H., 1984, High-temperature, large-volume, lavalike ash-flow tuffs without calderas in southwestern Idaho: U.S. Geological Survey Professional Paper, 1272, 76 p.

Elkin, D.C., 1993 (September), *Kinross Gold U.S.A. Inc. DeLamar Gold and Silver Mine Owhyhee County, Idaho Ore Reserves as of December 31, 1992*: unpublished report by Mine Reserves Associates, Inc. prepared for Kinross Gold Corporation, 39 p. plus appendices.

Gierzycki, G.A., 2004a (April), *Exploration potential of the DeLamar Mine property, Owyhee County, Idaho*: report prepared for Kinross Gold Corporation, 33 p.

Gierzycki, G.A., 2004b (November), *Deep gold-silver potential of the DeLamar mine property, Owyhee County, Idaho*: report prepared for Kinross Gold Corporation, 12 p.

Gray, J.N., Singh, R.B., Pennstrom Jr., W.J., Kunkel, K.W., and Cunningham-Dunlop, I. R., 2016 (January), *NI 43-101 Technical report and updated mineral resource estimate for the Castle Mountain project, San Bernardino County California, USA*: prepared for Newcastle Gold Ltd., 212 p.

Gustin, M.M., and Weiss, S.I., 2017 (November), *Technical Report and Resource Estimate, DeLamar Gold-Silver Project, Owyhee County, Idaho, USA*: NI 43-101 report prepared for Integra Resources Corp., 124 p.

Gustin, M.M., and Weiss, S.I., 2018 (March), *Technical Report and Resource Estimate for the DeLamar and Florida Mountain Gold-Silver Project, Owyhee County, Idaho, USA*: NI 43-101 report prepared for Integra Resources Corp., 154 p.

Gustin M.M., Weiss, S.I., and McPartland, J.S, 2019 (July), *Technical Report and Updated Resource Estimates for the DeLamar and Florida Mountain Gold-Silver Project, Owyhee County, Idaho, USA*: NI 43-101 report prepared for Integra Resources Corp., 207 p.

Halsor, S. P., 1983, *A volcanic dome complex and genetically associated hydrothermal system, DeLamar silver mine, Owyhee County, Idaho*: unpublished M.Sc. thesis, Michigan Tech. Univ., 111 p.

Halsor, S.P., Bornhorst, T.J., Beebe, M., Richardson, K., and Strowd, W., 1988, *Geology of the DeLamar silver mine, Idaho - A volcanic dome complex and associated hydrothermal system*: Economic Geology: v. 83, p. 1159–1169.

Hampton, P., 1988 (August), *Final Report on Phase-1 Florida Mt. Metallurgy*: internal company report, NERCO Minerals.

Hedenquist, J.W., 2018 (September), *Observations on gold-silver deposits of the DeLamar district, Idaho, and district potential beneath near-paleosurface outcrops*: unpublished report prepared for Integra Resources Corporation, 24 p.

Jordan, T., 2019 (March), *Sampling Narrative*: Internal memorandum prepared for DeLamar Mining Company.

Kilborn Engineering BC, 1988 (November), *Nerco Delamar Co. 'Floridan' Mountain Project – Review of Metallurgical Testwork*: report prepared for NERCO Minerals.

Lindberg, P.A., 1985 (September), *Geological appraisal of the Florida Mountain gold-silver district, Idaho*: unpublished report prepared for NERCO Minerals Company, 19 p. plus plates.

Lindgren, W., 1900, *The gold and silver veins of the Silver City, De Lamar, and other mining districts in Idaho*: U.S. Geological Survey 20th Annual Report, Part 3, p. 65–256.

Lindgren, W., and Drake, N.F., 1904, *Description of the Silver City quadrangle*: U.S. Geological Survey Geologic Atlas, Silver City Folio.

Mason, M.S., Saunders, J.A., Aseto, C., Hames, W.E., and Brueseke, M.E., 2015, *Epithermal Au-Ag ores of War Eagle and Florida Mountains, Silver City district, Owyhee County, Idaho*: in Pennell, W.M., and Garside, L.J., eds., Proceedings of the Geological Society of Nevada Symposium, New Concepts and Discoveries, Reno, p. 1067-1078.

McPartland, J.S., 2019a (June), *Summary Update Report on Metallurgical Testing– DeLamar 2018/2019 Samples*: report prepared for Integra Resources Corp. by McClelland Laboratories Inc., MLI Job No. 4307.

McPartland, J.S., 2019b (June), *Summary Update Report on Metallurgical Testing– Florida Mountain 2018/2019 Samples*: report prepared for Integra Resources Corp. by McClelland Laboratories Inc., MLI Job No. 4307.

McPartland, J.S., 2019c (August), *DeLamar Metallurgy Update*: memo prepared for Integra Resources Corp. by McClelland Laboratories Inc., MLI Job No. 4307.

Miyoshi, T.K., 1974 (January), *Preliminary metallurgical testing on the North DeLamar ore*: report prepared for Earth Resources Company by Hazen Research Inc., HRI Project 1466.

Miyoshi, T.K., and Light, R.H., 1974 (June), *Metallurgical Testing of a Composite Sample of North DeLamar Ore*: report prepared for Earth Resources Company by Hazen Research Inc., HRI Project 1520.

Miyoshi, T.K., Zaman, S., and Yarroll, W.H., 1971 (August), *Metallurgical and economic studies of a silver ore for Earth Resource Company*: report prepared by Hazen Research Inc., HRI Project No 979.

Mosser, K.L., 1992, *Mineralogy, paragenesis, and fluid inclusion relationships of the hydrothermal ore deposits at Florida Mountain, Carson mining district, Owyhee County, Idaho*: unpublished M.Sc. thesis, University of Arizona, 216 p.

Pancoast, L., 1990, *1989 South Wahl geologic model*: NERCO Exploration Company internal report, 9p plus appendices.

Panze, A.J., 1971, *Geology and ore deposits of the Silver City-De Larmar-Flint Creek region, Owyhee County, Idaho*: unpublished Ph.D. Thesis, Colorado School of Mines, 150 p.

Panze, A.J., 1972, *K-Ar ages of plutonism, volcanism and mineralization, Silver City region, Owyhee County, Idaho*: Isochron/West, no. 4, p. 1-4.

Pansze, A.J., 1975, *Geology and ore deposits of the Silver City-DeLamar-Flint region, Owyhee County, Idaho*: Idaho Bureau of Mines and Geology Pamphlet 161, 79 p.

Perry, J.K., 1971 (August), *Mineralogy of silver-bearing drill core from De Lamar, Idaho*: unpublished report prepared for Earth Resources Company by Hazen Research Inc., HRI Project No. 979, 35 p.

Piper, A.M., and Laney F.B., 1926, *Geology and metalliferous resources of the region about Silver City, Idaho*: Idaho Bureau of Mines and Geology Bulletin 11, 165 p.

Porterfield, B., 1992, Underground reserve potential at the DeLamar Mine: internal company report prepared for NERCO Minerals Company, 10 p.

Porterfield, B., and Moss, K., 1988 (March), *Geology and mineralization of Florida Mountain*: internal company report for NERCO Minerals Company, 31 p., plus plates.

Rak, P., Shaw, D.R., and Schmidt, R., 1989 (March), *Sullivan Gulch Gold Silver Ore – Metallurgical Studies*: report prepared for NERCO Minerals by Hazen Research Inc.

Richardson, K., 1985 (November), *Fire AA adjustment factors used to generate final silver values in computer data base*; internal NERCO memorandum, 4 p.

Rodgers B., 1980, *DeLamar silver mine, Owyhee County, Idaho*: unpublished company report prepared for Earth Resources Co., 6 p.

Sillitoe, R.H., 2018 (July), *Comment on geology and exploration of the DeLamar epithermal gold-silver district, Idaho*: unpublished report prepared for Integra Resources Corporation, 12 p.

Sillitoe, R.H., and Hedenquist, J.W., 2003, *Linkages between volcanotectonic settings, ore-fluid compositions, and epithermal precious metal deposits*: Soc. Economic Geologists Special Publication 10, p. 315-343.

Statter, D.J., 1989 (May), *Progress Report #8 on Florida Mountain Metallurgy*: internal NERCO company report, 2 volumes.

Taubeneck, W.H., 1971, *Idaho batholith and its southern extension*: Geological Society of America Bulletin, v. 82, p. 1899-1928.

Thomason, R. E., 1983, *Volcanic stratigraphy and epithermal mineralization of the DeLamar silver mine, Owyhee County, Idaho*: unpublished M.Sc. thesis, Oregon State Univ., 70 p.

Wells, W.W., 1963, *Gold camps and silver cities*: Idaho Bureau of Mines and Geology Bulletin 22, 36 p.

27.0 DATE AND SIGNATURE PAGE

Effective Date of report: September 9, 2019

Completion Date of report: October 22, 2019

"Michael M. Gustin"
Michael M. Gustin, C.P.G.
Date Signed: October 22, 2019

"Steven I. Weiss"
Steven I. Weiss, PhD, C.P.G.
Date Signed: October 22, 2019

"Thomas L. Dyer"
Thomas L. Dyer, P.E.
Date Signed: October 22, 2019

"Jack S. McPartland"
Jack S. McPartland, Member M.M.S.A.
Date Signed: October 22, 2019

"Jeffrey L. Woods"
Jeffrey L. Woods, Member S.M.E, M.M.S.A.
Date Signed: October 22, 2019

"John D. Welsh"
John D. Welsh, P.E.
Date Signed: October 22, 2019

28.0 CERTIFICATE OF QUALIFIED PERSONS

MICHAEL M. GUSTIN, C.P.G.

I, Michael M. Gustin, C.P.G., do hereby certify that I am currently employed as Senior Geologist by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Geology from Northeastern University in 1979 and a Doctor of Philosophy degree in Economic Geology from the University of Arizona in 1990. I have worked as a geologist in the mining industry for more than 30 years. I am a Licensed Professional Geologist in the state of Utah (#5541396-2250), a Licensed Geologist in the state of Washington (# 2297), a Registered Member of the Society of Mining Engineers (#4037854RM), and a Certified Professional Geologist of the American Institute of Professional Geologists (#CPG-11462).
2. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101"). I have previously explored, drilled, evaluated and modeled similar volcanic-hosted epithermal gold-silver deposits in the western US and Mexico. I certify that by reason of my education, affiliation with certified professional associations, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
3. I visited the DeLamar project site on August 16, 17, and 18, 2018.
4. Subject to those issues discussed in Section 3.0, I am responsible for Sections 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12 and 14 of this report titled, "*Technical Report and Preliminary Economic Assessment for the DeLamar and Florida Mountain Gold – Silver project, Owyhee County, Idaho, USA*", with an effective date of September 9, 2019 (the "Technical Report"). I am co-responsible for Sections 1, 24, 25 and 26 of this Technical Report.
5. I was a co-author of previous Technical Reports prepared for Integra Resources Corp. in 2017 and 2018, and assisted Kinross Gold Corporation with an evaluation of the project in 2016, but I am independent of Integra Resources Corp., and all of its subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
6. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report for which I am responsible for not misleading.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 22nd day of October 2019.

"Michael M. Gustin"

Michael M. Gustin

CERTIFICATE OF QUALIFIED PERSON

STEVEN I. WEISS, PH.D., C.P.G.

I, Steven I. Weiss, C.P.G., do hereby certify that:

- I am currently a self-employed Senior Associate Geologist for Mine Development Associates, Inc., located at 210 South Rock Blvd., Reno, Nevada, 89502; and
- I graduated with a Bachelor of Arts degree in Geology from the Colorado College in 1978, received a Master of Science degree in Geological Science from the Mackay School of Mines at the University of Nevada, Reno in 1987, and hold a Doctorate in Geological Science from the University of Nevada, Reno, received in 1996.
- I am a Certified Professional Geologist (#10829) with the American Institute of Professional Geologists and have worked as a geologist in the mining industry and in academia for more than 35 years.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101"). I have previously explored, drilled, evaluated and reported on gold-silver deposits in volcanic and sedimentary rocks in Nevada, California, Canada, Greece, and Mexico. I certify that by reason of my education, affiliation with certified professional associations, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am a co-author of this Technical Report titled *"Technical Report and Preliminary Economic Assessment for the DeLamar and Florida Mountain Gold – Silver project, Owyhee County, Idaho, USA"* prepared for Integra Resources Corp., and with an effective date of September 9, 2019. Subject to those issues discussed in Section 3.0, I am co-responsible for Sections 1, 2, 3, 5, 6, 7, 8, 9, 10, 11, 24, 25, and 26 of this Technical Report.
- I was a co-author of previous Technical Reports prepared for Integra Resources Corp. in 2017 and 2018, but prior to this I have not had involvement with the property that is the subject of this Technical Report. I visited the DeLamar project site on August 1st, 2nd and 3rd, 2017.
- To the best of my knowledge, information and belief, as of the effective date the Technical Report contains the necessary scientific and technical information to make the Technical Report not misleading.
- I am independent of Integra Resources Corp., and all of their respective subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
- I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with the requirements of that instrument and form.

Dated this 22nd day of October 2019

"Steven I. Weiss"

Signature of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

THOMAS L. DYER, P.E.

I, Thomas L. Dyer, P.E., do hereby certify that:

(1) I am currently employed as Senior Engineer at Mine Development Associates, whose address is 210 S. Rock Blvd., Reno, NV 89502.

(2) I am a co-author of this Technical Report titled *"Technical Report and Preliminary Economic Assessment for the DeLamar and Florida Mountain Gold – Silver project, Owyhee County, Idaho, USA"* prepared for Integra Resources Corp., and with an effective date of September 9, 2019.

(3) I graduated with a Bachelor of Science degree in Mine Engineering from South Dakota School of Mines and Technology in 1996. I am a Registered Professional Engineer in the state of Nevada (#15729) and a Registered Member (#4029995RM) of the Society of Mining, Metallurgy and Exploration.

(4) I have worked as a mining engineer for more than 22 years since my graduation. Relevant experience includes providing mine designs, reserve estimates and economic analyses of precious-metals deposits and industrial minerals deposits in the United States and various countries of the world. During this period I have worked as Chief Engineer of an operating heap leach and mill gold mine in Nevada.

(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 have not visited Florida Mountain or DeLamar.

(7) I take responsibility for Sections 15, 16, 18, 19, 21 (except for 21.2 and 21.6) and Section 22 of this report, subject to those issues discussed in Section 3. I take joint responsibility for Sections 1, 24, and 25.

(8) I am independent of Integra Resources Corp., and all of their respective subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

(9) I have had no prior involvement with the property that is the subject of this report.

(10) I have read National Instrument 43-101 and those portions of this report for which I am responsible have been prepared in compliance with that Instrument.

(11) As of the effective date of the technical report, to the best of my knowledge, information, and belief, the technical report, or part that I am responsible for, contains all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 22nd day of October 2019

"Thomas L. Dyer

Signature of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

JACK S. McPARTLAND, METALLURGIST/PRESIDENT

I, Jack McPartland, do hereby certify that I am currently employed as Metallurgist/President, McClelland Laboratories, Inc., 1016 Greg Street, Sparks, Nevada 89431, and:

1. I graduated with a Bachelor of Science degree in Chemical Engineering from the University of Nevada, Reno in 1986 and a Master of Science degree in Metallurgical Engineering from the University of Nevada, Reno in 1989. I have worked as a metallurgist for a total of 30 years since my graduation from undergraduate university, managing and evaluating metallurgical testing and designing mineral processing systems for numerous base-metal and precious-metal mining projects in North and South America.

2. I am a registered member of the Mining and Metallurgical Society of America, and I am recognized as a Qualified Professional (QP) Member with special expertise in Metallurgy/Processing (Member No. 01350QP).

3. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

4. I visited the DeLamar project site on January 17, 2019. I am responsible for Item 1.5 and Item 13 of this report titled, "*Technical Report and Preliminary Economic Assessment for the DeLamar and Florida Mountain Gold – Silver project, Owyhee County, Idaho, USA*", with an effective date of September 9, 2019 (the "Technical Report").

5. I have had no prior involvement with the property that is the subject of the Technical Report and I am independent of Integra Resources Corp., and all of its subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

6. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report for which I am responsible for not misleading.

7. 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.

Dated this 22nd day of October 2019

"Jack S. McPartland"
Signature of Qualified Person

CERTIFICATE OF QUALIFIED PERSONS

JEFFREY L. WOODS, S.M.E, M.M.S.A.

I, Jeffrey L. Woods, SME MMSA, do hereby certify that I am employed as the Principle Consulting Metallurgist with Woods Process Services, LLC located at 319 Quitman St., Denver CO and:

1 I am a member in good standing of Society for Mining, Metallurgy and Exploration, membership #4018591.

2 I graduated from the Mackay School of Mines, University of Nevada, Reno, Nevada, U.S.A., in 1988 with a B.S. in Metallurgical Engineering.

3 I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101"). I have been directly involved in international mine operations, technical services, project development and consulting for various commodities, metals, deposits and processes. As a result of my experience and qualifications, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have practiced my profession continuously for 31years since graduation.

5 I am responsible for Section 17, the processing component of Section 21 and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those sections in this Technical Report titled *"Technical Report and Preliminary Economic Assessment for the DeLamar and Florida Mountain Gold – Silver project, Owyhee County, Idaho, USA"* prepared for Integra Resources Corp., and with an effective date of September 9, 2019. .

6 I have not visited the DeLamar project site as of the effective date of this Technical Report.

7 I am independent of Integra Resources as described in the instrument.

8 As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report for which I am responsible for not misleading.

9 I have read NI 43–101 and the Technical Report sections for which I am responsible have been prepared in compliance with that Instrument.

Dated this 22nd day of October 2019

"JL Woods"

Jeffrey L. Woods

CERTIFICATE OF QUALIFIED PERSONS

JOHN D. WELSH, P.E.

I, John D. Welsh, P.E. do hereby certify that I am currently employed as a Senior Principal Engineer by Welsh Hagen Associates, Inc., 250 South Rock Street, Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Civil Engineering from the University of Missouri at Rolla in 1970 and a Masters of Science degree in Civil (Geotechnical) Engineering from Colorado State University in 1979. I have worked as a geotechnical engineer in the mining industry for 45 years. I am a Registered Professional Civil Engineer in the state of Nevada (#6296) and the state of California (#35861). I am a member of the Society of Mining Engineers.
2. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and I have previously performed similar designs for mining facilities in Idaho, Nevada, Colorado, and California. I certify by reason of my education, professional certifications, and relevant work experience that I fulfill the requirements of a "qualified person" for the purposes of NI 43-101.
3. I last visited the property on June 26, 2019.
4. I am co-responsible for Sections 1, 18, 21, and 25 of this report titled, *Technical Report and Preliminary Economic Assessment for the DeLamar and Florida Mountain Gold-Silver Project, Owyhee county, Idaho, USA, with an effective date of September 9, 2019 (the "Technical Report").*
5. I am independent of Integra Resources Corp., and all of its subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
6. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report for which I am responsible not misleading.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8. Dated this 22nd day of October 2019

John D. Welsh

John D. Welsh

APPENDIX A

LISTING OF PATENTED AND UNPATENTED FEDERAL MINING CLAIMS AND LEASED LAND

Part 1: Owned and Leased Patented Claims, One Leased Unpatented Claim and Leased State of Idaho Lands

Owned Real Property (Owyhee County, ID):

Patented Mining Claims

1.0 TAX PARCEL #RP 95S04W050106A

LODES:

BOSTON, MS 855; CASH, MS 859A; CHICAGO, MS 643A; CHRISTIAN WAHL, MS 642A; CROWN PRINCE & BISMARCK CONSOLIDATED, MS 923A; DENVER, MS 856A; DISSON, MS 921; HIDDEN TREASURE, MS 1264; HOPE, MS 920A; IBURG, MS 1260; IDAHO, MS 548; LONDON, MS 857A; LOUIS WAHL, MS 854; MICHIGAN, MS 1266; MOLLOY, MS 1029A; NEW YORK, MS 863A; PHEBE GRACE, MS 858; PHILADELPHIA, MS 862A; SAN FRANCISCO, MS 860; STODDARD, MS 38; TORPEDO, MS 1261; WALLSTREET, MS 1265; WILSON, MS 547; ZULU, MS 1259.

MILLSITES:

CASH MILL SITE, MS 859B; CHICAGO MILL SITE, MS 643B; CHRISTIAN WAHL MILL SITE, MS 642B; CROWN PRINCE & BISMARCK CONSOLIDATED, MS 923B; DELAMAR MILL SITE, MS 1024; DENVER MILL SITE, MS 856B; HOPE MILL SITE, MS 920B; LONDON MILL SITE, MS 857B; NEW YORK MILL SITE, MS 863B; PHILADELPHIA MILL SITE, MS 862B; WILSON MILL SITE, MS 652.

2.0 TAX PARCEL #RP 95S04W060146A

Leply group, MS 3066, ADVANCE, BOONE, CHATAQUA (sic), INDEPENDENCE, and a portion of BECK and LAST CHANCE

3.0 TAX PARCEL #RP 95S04W050147A

BECK, LAST CHANCE, MS 3066, described as Lot 47.
Per Assessor's office, said Lot 147 is a portion of Beck and Last Chance (Leply group)

4.0 TAX PARCEL #RP 95S04W08119AA

PORTION OF IBURG, MS 1260, Tax 119A

5.0 TAX PARCEL #RP 95S04W050151A

ELLA, CZARINA, ONLY CHANCE, BADGER, MS 3067

6.0 TAX PARCEL #RP 95S04W05074AA

HOWE, MS 950A, & MANHATTAN, MS 866, less a portion

7.0 TAX PARCEL #RP 95S04W05074BA

PORTION OF HOWE, MS 950A, & MANHATTAN, MS 866

8.0 Tax parcel #RP 95S04W056000A

NDCO SEC5 #27, 28, [29-32], 30, 31, [34-35], 36, 37, 38, 39, 40

9.0 TAX PARCEL #RP 95S04W068400A

NDCO SEC6 #17, 18, 19, 20, 21, 22, 23, 24, 25, 29, 33, 34, 35, 36, 37, 38, 39, 40, 41, 42, 43

10.0 Tax parcel #RP 95S04W072300A

NDCO SEC7 #6, 7, 8, 9, 10, 11, 12, 13, 14, 15, 30, 31, 32, 33, 34, 35, 36, 37, 38, 39, 40, 41

11.0 TAX PARCEL #RP 95S04W084300A

NDCO SEC8 #8, 9, 10, 11, 12, 13, 14, 15, 16, 17, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27, 28, 29, 30, 31, 32, 33, 34,
35, 36, 37, 38, 39, 40, 41, 42, 43, 44, 45, 46, 47, 48, 49, 50, 51, 52, 53, 54, 55, 56, 57, 58, 59, 60, 61, 62, 63, 64, 65,
66, 67, 68, 69, 70, 73, 74, 75, 76, 77, 78, 79, 80, 81, 82, 83, 84, 85, 87, 88, 89, 90

12.0 TAX PARCEL #RP 95S04W094600A

NDCO SEC9 #8, 9, 10, 11, 12, 13, 14, 15, 16, 17, 18, 19, 20, 21, 22, 23, 24, 25, 35, 36, 37, 38, 39, 40, 41, 42, 43,
44, 45, 46, 47, 48, 50, 51, 52, 53, 54, 55, 56, 57

13.0 Tax parcel #RP 95S04W01093FA EUREKA, MS 3100, located in 1-5S-4W

14 & 15 Tax parcel #RP 95S04W01057AA and Tax parcel #RP 95S04W01043AA

Banner, Harmon, C.O.D, Mammon, Ella, Coffee, Star Spangle, Tip Top, Justice, Apex

16.0 TAX PARCEL #RP 95S04W01001AA

BLACK JACK, EMPIRE STATE, PHILLIPS, SULLIVAN, BELFAST, COLORADO, SIERRA NEVADA, INDEPENDENCE, JUMBO, SOUTH PLUTO, BLACK BART, JAMES C. BLAINE, TRADE DOLLAR, FRACTION, SOUTH EXTENSION, BLAINE, CAROLINE, OWYHEE TREASURY, SEVENTY NINE, J.M. GUFFY, ALPINE, LITTLE CHIEF, HARRISON, ALLEGHANY, TWENTY ONE, SUNFLOWER, INDUSTRY, ECONOMY, NORTH EXTENSION, COMMONWEALTH, ROUGH AND READY, COMMONWEALTH, COMSTOCK, BALTIC, STERLING, BLACK JACK MILLSITE, PLUTO MILLSITE, PALM BEACH INN

17.0 Leased Patented Claims (Owyhee County, ID):

A. Elordi

Description: HENRIETTA, MS#630, Patent #17275, in Sec 6, T5S, R4W, BM Royalty: 5% NSR until $50,000 has been paid, thereafter 2.5% NSR until $400,000 paid.

B. Getchell/Gross

Description: OHIO, MS #3064, Patent #1031892, in Sec. 4, T5S, R4W, BM
Royalty: 5% NSR

C. Elordi

Description: MAMMOTH & ANACONDA, MS 2151, Patent #45359, Sec 1&2, T5S, R4W, BM
Royalty: 2.5% NSR

D. Brunzell/Jayo/Brunzell

Description: SUMMIT, MS#2383, Patent #88744, in Sec 1, T5S, R4W, BM
Royalty: 2.5% NSR

E. Statham

Description: The following 9 patented claims and 1 unpatented claim in Sec 31, T4S, R3W; Sec 36, T4S,R4W; Sec 1, T5S,R4W; Sec 6, T5S,R3W, BM

Royalty: 2.5% NSR to a maximum of $650,000.

Unpatented Claim Name	BLM No.
The Holy Terror Placer No. 1 Placer Claim	IMC # 23906

Patented Claims

Survey Number	Patent No.	Claim Name
MS 2155	54089	September
MS 1913	40635	Joseph
MS 1909	40636	True Blue
MS 1910	40637	George Washington
MS 1908	40637	Palmer
MS 1906	40637	Eagle
MS 1912	40637	Kentuck
MS 1907	40637	Eclipse
MS 1911	40637	North Extension Humboldt

F. Nottingham

Description: The following 12 patented claims in Sec 1 & 2, T5S,R4W, BM

Royalty: 2% NSR to a maximum of $400,000

Survey Number	Patent Number	Claim Name
MS 3101	1019060	Alright
MS 3100	1019061	Eureka (7.3 acres)
MS 3100	1019061	Search Light
MS 1968A	44196	Harrison
MS 1968A	44196	Blaine
MS 1968A	44196	Shannon
MS 1968A	44196	Molly Pincher
MS 1968A	44196	Tonowanda Placer
MS 3103	1019062	Roosevelt Placer
MS 3099	1019063	Ida May
MS 3099	1019063	Nellie Grant
MS 3102	1019059	King Edward

Leased Lands (Owyhee County, ID):

Six State of Idaho Department of Lands Leases

	Lease No.	Acreage	Description	Status
1	E600067	396	T4S,R5W,S.36	Issued
2	E600085	640	T4S,R5W,S.36	Pending
3	E600086	601	T4S,R5W,S.35; T5S,R5W,1	Pending
4	E600090	640	T5S,R4W,S.16	Pending
5	E600092	514	T4S,R4W,S.31	Pending
6	E600093	557	T4S,R4W,S.28,29,33; T5S,R4W,S7	Pending

Part 2: 284 Unpatented Claims Owned or Controlled by DeLamar Mining Co.

Claim #	Claim Name	BLM #
1 *(160 ac Placer)*	Barnes	IMC-50576
2 *(38 ac Placer)*	Blue Gulch	IMC-50577
3	Century	IMC-19303
4 *160 ac Placer)*	CHINA	IMC-49020
5	COLUMBIA	IMC-19297
6	Cook 2	IMC-16257
7	Cook 3	IMC-16258
8	Cook 6	IMC-16261
9	Cook 8	IMC-16263
10	Cook 10	IMC-16265
11	Cook 12	IMC-16267
12	Cook 14	IMC-16269
13	Cook 16	IMC-16271
14	Cook 19	IMC-16274
15	Cook 48	IMC-16303
16	Cook 52	IMC-16307
17	Cook 53	IMC-16308
18	Cook 54	IMC-16309
19	Cook 56	IMC-16311
20	Cook 57	IMC-16312
21	Cook 58	IMC-16313
22	Cook 60	IMC-16315
23	Cook 62	IMC-16317
24	Cook 74	IMC-16329
25	Cook 75	IMC-16330
26	Cook 76	IMC-16331
27	Cook 77	IMC-16332
28	Cook 78	IMC-16333
29	Cook 79	IMC-16334
30	Cop 1	IMC-16337
31	Cop 3	IMC-16339
32	Cop 5	IMC-16341
33	Cop 7	IMC-16343
34	Cop 9	IMC-16345
35	Cop 11	IMC-16347
36	Cop 13	IMC-16349
37	Cop 15	IMC-16351
38	Cop 17	IMC-16353
39	Cop 19	IMC-16355
40	Cop 21	IMC-16357

Claim #	Claim Name	BLM #
41	Cop 22	IMC-16358
42	Cop 23	IMC-16359
43	Cop 24	IMC-16360
44	Cop 25	IMC-16361
45	Cop 26	IMC-16362
46	Cop 32	IMC-16368
47	Cop 33	IMC-16369
48	Cop 34	IMC-16370
49	Cop 35	IMC-16371
50	Cop 40	IMC-16376
51	Cop 68	IMC-16404
52	Cop 69	IMC-16405
53	Cop 70	IMC-16406
54	Cop 73	IMC-16409
55	Cop 74	IMC-16410
56	Cop 75	IMC-16411
57	Cop 78	IMC-16414
58	Cop 80	IMC-16416
59	DALY	IMC-20390
60	DAM #8	IMC-136064
61	DAM #12	IMC-136068
62	DAM #13	IMC-136069
63	DAM #28	IMC-136072
64	DELAGARDE	IMC-19299
65	DeLamar #5 Fraction	IMC-11235
66	DeLamar Fraction #1A	IMC-11231
67	DeLamar Fraction #6	IMC-11236
68	DeLamar Fraction #7	IMC-11237
69	DeLamar Fraction #9	IMC-13720
70	DeLamar Fraction #11	IMC-13722
71	DeLamar Fraction #13	IMC-11239
72	DeLamar Fraction #14	IMC-13724
73	DeLamar Fraction #15	IMC-11240
74	DeLamar Fraction #16	IMC-11241
75	DeLamar Fraction #20	IMC-50823
76	DeLamar Fraction 2A	IMC-11232
77	DeLamar Fraction 3A	IMC-11233
78	DeLamar Fraction 4	IMC-11234
79	DeLamar Fraction 17	IMC-11242
80	DeLamar Fraction 18	IMC-11243
81	DeLamar Fraction 19	IMC-50822
82	DeLamar Fraction 19A	IMC-11244
83	DeLamar Fraction 20	IMC-11245

Claim #	Claim Name	BLM #
84	DeLamar Fraction 21	IMC-50824
85	DL-2	IMC-217429
86	DL-3	IMC-217430
87	DL-4	IMC-217431
88	DL-5	IMC-217432
89	DL-6	IMC-217433
90	DL-7	IMC-217434
91	DL-8	IMC-217435
92	DL-9	IMC-217436
93	DL-10	IMC-217437
94	DL-11	IMC-217438
95	DL-12	IMC-217439
96	DL-13	IMC-217440
97	DL-14	IMC-217441
98	DL-15	IMC-217442
99	DL-16	IMC-217443
100	DL-17	IMC-217444
101	DLF #36	IMC-153395
102	DLF-23	IMC-65556
103	DLF-24	IMC-65557
104	DLF-25	IMC-65558
105	DLF-26	IMC-65559
106	DLF-27	IMC-65560
107	DLF-28	IMC-65561
108	DLF-29	IMC-65562
109	DLF-30	IMC-65563
110	DLF 33	IMC-134646
111	DLF 34	IMC-134647
112	DLF 35	IMC-134648
113	Elko	IMC-13655
114	Elko No.2	IMC-13656
115	ENGL 1	IMC-14687
116	ENGL 2	IMC-137927
117	ENGL 3	IMC-14689
118	ENGL 4	IMC-14690
119	ENGL 5	IMC-14691
120	ENGL 6	IMC-137928
121	ENGL 7	IMC-137929
122	ENGL 7A	IMC-137930
123	ENGL 8	IMC-163888
124	ENGL 9	IMC-16228
125	ENGL 10	IMC-16229
126	ENGL 11	IMC-16230

Claim #	Claim Name	BLM #
127	ENGL 12	IMC-16231
128	ENGL 13	IMC-16232
129	ENGL 14	IMC-16233
130	ENGL 15	IMC-16234
131	ENGL 16	IMC-16235
132	ENGL 17	IMC-16236
133	ENGL 19	IMC-16238
134	ENGL 21	IMC-16240
135	ENGL 23	IMC-163889
136	ENGL 24	IMC-16243
137	ENGL 25	IMC-16244
138	ENGL 26	IMC-16245
139	ENGL 27	IMC-16246
140	ENGL 28	IMC-16247
141	ENGL 29	IMC-16248
142	ENGL 30	IMC-16249
143	ENGL 31	IMC-16250
144	ENGL 32	IMC-16251
145	ENGL 33	IMC-16252
146	ENGL 34	IMC-16253
147	ENGL 35	IMC-16254
148	ENGL 36	IMC-16255
149	FM-1 Fraction	IMC-11485
150	FM 16-Fraction	IMC-111724
151	FM 18-Fraction	IMC-111726
152	FM 19-Fraction	IMC-111727
153	FM 20-Fraction	IMC-111728
154	FM 21-Fraction	IMC-111729
155	FM 22 Fraction	IMC-111730
156	FM 23 Fraction	IMC-111731
157	FM Fraction #2	IMC-11486
158	FM Fraction #3	IMC-11487
159	FM Fraction #5	IMC-11489
160	FM Fraction #6	IMC-11490
161	FM Fraction #7	IMC-11491
162	FM Fraction #8	IMC-11492
163	FM Fraction #9	IMC-11493
164	FM Fraction #10	IMC-11494
165	FMP-4	IMC-125864
166	FMP-5	IMC-125865
167	FMP-6	IMC-125866
168	FMP-7	IMC-125867
169	FMP-12	IMC-125872

Claim #	Claim Name	BLM #
170	FMP-13	IMC-125873
171	FMP-14	IMC-125874
172	FMP-15	IMC-125875
173	FMP-21	IMC-125882
174	GLOBE	IMC-20389
175	Golden Gate	IMC-19300
176	Gold Standard #4	IMC-13714
177	Grand Central	IMC-20391
178	GS-1	IMC-13672
179	GS-2	IMC-13673
180	GS-3	IMC-13674
181	GS-4	IMC-13675
182	GS-5	IMC-13676
183	GS-6	IMC-13677
184	GS-7	IMC-13678
185	GS-9	IMC-13680
186	GS-11	IMC-13682
187	GS-13	IMC-13684
188	GS-14	IMC-13685
189	GS-15	IMC-13686
190	GS-16	IMC-13687
191	GS-17	IMC-13688
192	GS-26	IMC-13697
193	GS-27	IMC-13698
194	Hawk #1	IMC-1043
195	Hawk *#2*	IMC-1044
196 *(160 ac Placer)*	JACOBS	IMC-49021
197	LAST CHANCE	IMC-19298
198 *(160 ac placer)*	LAST CHANCE	IMC-50579
199	Little Rose	IMC-19293
200	M&D	IMC-169336
201	MARY LYNN 1	IMC-163890
202	MARY LYNN 2	IMC-163891
203	MARY LYNN 3	IMC-163892
204	MARY LYNN 4	IMC-163893
205 *(160 ac Placer)*	MERCURY	IMC-50578
206	MONO	IMC-19294
207	MS-1	IMC-217422
208	MS-2	IMC-217423
209	MS-3	IMC-217424
210	MS-4	IMC-217425
211	MS-5	IMC-217426
212	MS-6	IMC-217427

Claim #	Claim Name	BLM #
213	MS-7	IMC-217428
214	MVC	IMC-169335
215	New Deal	IMC-19301
216	Noon Silver	IMC-13703
217	North Chance	IMC-13705
218	North DeLamar #4	IMC-13728
219	North DeLamar #7	IMC-13731
220	NORTHERN LIGHT	IMC-19295
221	North Summit	IMC-13709
222	Ontario	IMC-11500
223	PAYETTE	IMC-20392
224	Progress	IMC-19302
225	Rawhide A	IMC-13716
226	Red Cloud	IMC-14797
227	RG 1	IMC-140230
228	RG 3	IMC-140232
229	RG 5	IMC-140234
230	RG 7	IMC-140236
231	RG 41	IMC-140270
232	RG 43	IMC-140272
233	RG 56	IMC-140285
234	RG 57	IMC-140286
235	RG 58	IMC-140287
236	RG 59	IMC-140288
237	SC 5	IMC-160973
238	SC 6	IMC-160974
239	SC 7	IMC-160975
240	SC 10	IMC-160978
241	SKYLARK	IMC-19296
242	South DeLamar #11	IMC-11259
243	South DeLamar #11A	IMC-11260
244	South DeLamar #12	IMC-11262
245	South DeLamar #12A	IMC-11261
246	South DeLamar #13	IMC-11263
247	South DeLamar #14	IMC-11264
248	South DeLamar #16	IMC-11266
249	South DeLamar #18	IMC-11268
250	South DeLamar #54A	IMC-167689
251	South DeLamar #55	IMC-61553
252	South DeLamar #56	IMC-61554
253	South DeLamar #57	IMC-61555
254	South DeLamar #58	IMC-61556
255	South DeLamar # 59	IMC-61557

Claim #	Claim Name	BLM #
256	South DeLamar #63	IMC-61561
257	South DeLamar No. 39	IMC-79
258	South DeLamar No. 40	IMC-80
259	South DeLamar No. 41	IMC-81
260	South DeLamar No. 42	IMC-844
261	South DeLamar No. 43	IMC-845
262	South DeLamar No. 48	IMC-850
263	South DeLamar No. 49	IMC-851
264	Summercamp A	IMC-13717
265	Summit	IMC-13704
266	Vein Dike	IMC-20388
267	Vein Dyke Fraction	IMC-20387
268	Virginia	IMC-11499
269 (160 ac Placer)	WAGON 1	IMC-49023
270 (160 ac Placer)	WAGON 2	IMC-49024
271	West Henrietta #2	IMC-53365
272	West Henrietta #3	IMC-53366
273	West Henrietta #4	IMC-53367
274	West Henrietta #5	IMC-53368
275	West Henrietta #6	IMC-53369
276	West Henrietta 7	IMC-53370
277	West Henrietta 8	IMC-53371
278	West Henrietta 9	IMC-53372
279	West Henrietta 10	IMC-53373
280	West Henrietta-11	IMC-53374
281	West Henrietta-12	IMC-53375
282	West Henrietta-13	IMC-53376
283	West Henrietta-15	IMC-53378
284	West Henrietta-16	IMC-53379

Part 3: 226 Unpatented Lode Claims Owned or Controlled by DeLamar Mining Co.

Claim #	Claim Name	BLM #
1	JG-1	IMC-221535
2	JG-2	IMC-221536
3	JG-3	IMC-221537
4	JG-4	IMC-221538
5	JG-5	IMC-221539
6	JG-6	IMC-221540
7	JG-7	IMC-221541
8	JG-8	IMC-221542
9	JG-9	IMC-221543
10	JG-10	IMC-221544
11	JG-11	IMC-221545
12	JG-12	IMC-221546
13	JG-13	IMC-221547
14	JG-14	IMC-221548
15	JG-15	IMC-221549
16	JG-16	IMC-221550
17	JG-21	IMC-221551
18	JG-22	IMC-221552
19	JG-23	IMC-221553
20	JG-24	IMC-221554
21	JG-25	IMC-221555
22	JG-26	IMC-221556
23	JG-27	IMC-221557
24	JG-28	IMC-221558
25	JG-29	IMC-221559
26	JG-30	IMC-221560
27	JG-31	IMC-221561
28	JG-32	IMC-221562
29	JG-33	IMC-221563
30	JG-34	IMC-221564
31	JG-35	IMC-221565
32	JG-36	IMC-221566
33	JG-37	IMC-221567
34	JG-38	IMC-221568
35	JG-39	IMC-221569
36	JG-40	IMC-221570
37	JG-41	IMC-221571
38	JG-42	IMC-221572
39	JG-43	IMC-221573
40	JG-44	IMC-221574
41	JG-45	IMC-221575

Claim #	Claim Name	BLM #
42	JG-46	IMC-221576
43	JG-47	IMC-221577
44	JG-48	IMC-221578
45	JG-49	IMC-221579
46	JG-50	IMC-221580
47	JG-51	IMC-221581
48	JG-52	IMC-221582
49	JG-53	IMC-221583
50	JG-54	IMC-221584
51	JG-55	IMC-221585
52	JG-56	IMC-221586
53	JG-57	IMC-221587
54	JG-58	IMC-221588
55	JG-59	IMC-221589
56	JG-60	IMC-221590
57	JG-61	IMC-221591
58	JG-62	IMC-221592
59	JG-63	IMC-221593
60	JG-64	IMC-221594
61	JG-65	IMC-221595
62	JG-66	IMC-221596
63	JG-67	IMC-221597
64	JG-68	IMC-221598
65	JG-69	IMC-221599
66	JG-70	IMC-221600
67	JG-71	IMC-221601
68	JG-72	IMC-221602
69	JG-73	IMC-221603
70	JG-74	IMC-221604
71	JG-75	IMC-221605
72	JG-76	IMC-221606
73	JG-77	IMC-221607
74	JG-78	IMC-221608
75	FMS-1	IMC-223228
76	FMS-2	IMC-223229
77	FMS-3	IMC-223230
78	FMS-4	IMC-223231
79	FMS-5	IMC-223232
80	FMS-6	IMC-223233
81	FMS-7	IMC-223234
82	FMS-8	IMC-223235
83	FMS-9	IMC-223236
84	FMS-10	IMC-223237

Claim #	Claim Name	BLM #
85	FMS-11	IMC-223238
86	FMS-12	IMC-223239
87	FMS-13	IMC-223240
88	FMS-14	IMC-223241
89	FMS-15	IMC-223242
90	FMS-16	IMC-223243
91	FMS-17	IMC-223244
92	FMS-18	IMC-223245
93	FMS-19	IMC-223246
94	FMS-20	IMC-223247
95	FMS-21	IMC-223248
96	FMS-22	IMC-223249
97	JG-79	IMC-223250
98	JG-80	IMC-223251
99	JG-81	IMC-223252
100	JG-82	IMC-223253
101	JG-83	IMC-223254
102	JG-84	IMC-223255
103	JG-85	IMC-223256
104	JG-86	IMC-223257
105	JG-87	IMC-223258
106	JG-88	IMC-223259
107	JG-89	IMC-223260
108	JG-90	IMC-223261
109	JG-91	IMC-223262
110	JG-92	IMC-223263
111	JG-93	IMC-223264
112	JG-94	IMC-223265
113	JG-95	IMC-223266
114	JG-96	IMC-223267
115	JG-97	IMC-223268
116	JG-98	IMC-223269
117	JG-99	IMC-223270
118	JG-100	IMC-223271
119	JG-101	IMC-223272
120	JG-102	IMC-223273
121	JG-103	IMC-223274
122	JG-104	IMC-223275
123	JG-105	IMC-223276
124	JG-106	IMC-223277
125	JG-107	IMC-224111
126	JG-108	IMC-224112
127	JG-109	IMC-224113

Claim #	Claim Name	BLM #
128	JG-110	IMC-224114
129	JG-111	IMC-224115
130	JG-112	IMC-224116
131	JG-113	IMC-224117
132	JG-114	IMC-224118
133	JG-115	IMC-224119
134	JG-116	IMC-224120
135	JG-117	IMC-224121
136	JG-118	IMC-224122
137	JG-119	IMC-224123
138	JG-120	IMC-224124
139	JG-121	IMC-224125
140	JG-122	IMC-224126
141	JG-123	IMC-224127
142	JG-124	IMC-224128
143	JG-125	IMC-224129
144	JG-126	IMC-224130
145	JG-127	IMC-224131
146	JG-128	IMC-224132
147	JG-129	IMC-224133
148	JG-130	IMC-224134
149	JG-131	IMC-224135
150	JG-132	IMC-224136
151	JG-133	IMC-224137
152	JG-134	IMC-224138
153	JG-135	IMC-224139
154	FMS-23	IMC-224140
155	FMS-24	IMC-224141
156	FMS-25	IMC-224142
157	FMS-26	IMC-224143
158	FMS-27	IMC-224144
159	FMS-28	IMC-224145
160	FMS-29	IMC-224146
161	FMS-30	IMC-224147
162	FMS-31	IMC-224148
163	FMS-32	IMC-224149
164	FMS-33	IMC-224150
165	FMS-34	IMC-224151
166	FMS-35	IMC-224152
167	FMS-36	IMC-224153
168	FMS-37	IMC-224154
169	FMS-38	IMC-224155
170	FMS-39	IMC-224156

Claim #	Claim Name	BLM #
171	FMS-40	IMC-224157
172	FMS-41	IMC-224158
173	FMS-42	IMC-224159
174	FMS-43	IMC-224160
175	FMS-44	IMC-224161
176	FMS-45	IMC-224162
177	FMS-46	IMC-224163
178	FMS-47	IMC-224164
179	FMS-48	IMC-224165
180	FMS-49	IMC-224166
181	FMS-50	IMC-224167
182	FMS-51	IMC-224168
183	FMS-52	IMC-224169
184	FMS-53	IMC-224170
185	FMS-54	IMC-224171
186	FMS-55	IMC-224172
187	FMS-56	IMC-224173
188	FMS-57	IMC-224174
189	FMS-58	IMC-224175
190	FMS-59	IMC-224176
191	FMS-60	IMC-224177
192	FMS-61	IMC-224178
193	FMS-62	IMC-224179
194	FMS-63	IMC-224180
195	FMS-64	IMC-224181
196	FMS-65	IMC-224182
197	FMS-66	IMC-224183
198	FMS-67	IMC-224184
199	FMS-68	IMC-224185
200	FMS-69	IMC-224186
201	FMS-70	IMC-224187
202	FMS-71	IMC-224188
203	FMS-72	IMC-224189
204	FMS-73	IMC-224190
205	FMS-74	IMC-224191
206	FMS-75	IMC-224192
207	FMS-76	IMC-224193
208	FMS-77	IMC-224194
209	FMS-78	IMC-224195
210	FMS-79	IMC-224196
211	FMS-80	IMC-224197
212	FMS-81	IMC-224198
213	FMS-82	IMC-224199

Claim #	Claim Name	BLM #
214	FMS-83	IMC-224200
215	FMS-84	IMC-224201
216	FMS-85	IMC-224202
217	FMS-86	IMC-224203
218	FMS-87	IMC-224204
219	FMS-88	IMC-224205
220	FMS-89	IMC-224206
221	FMS-90	IMC-224207
222	FMS-91	IMC-224208
223	FMS-92	IMC-224209
224	FMS-93	IMC-224210
225	FMS-94	IMC-224211
226	FMS-95	IMC-224212

Part 4: 165 Unpatented Lode Claims Owned or Controlled by DeLamar Mining Co.

Claim #	Claim Name	BLM #
1	JK 1	IMC 228627
2	JK 2	IMC 228628
3	JK 3	IMC 228629
4	JK 4	IMC 228630
5	JK 5	IMC 228631
6	JK 6	IMC 228632
7	JK 7	IMC 228633
8	JK 8	IMC 228634
9	JK 9	IMC 228635
10	JK 10	IMC 228636
11	JK 11	IMC 228637
12	JK 12	IMC 228638
13	JK 13	IMC 228639
14	JK 14	IMC 228640
15	JK 15	IMC 228641
16	JK 16	IMC 228642
17	JK 17	IMC 228643
18	JK 18	IMC 228644
19	JK 19	IMC 228645
20	JK 20	IMC 228646
21	JK 21	IMC 228647
22	JK 22	IMC 228648
23	JK 23	IMC 228649
24	JK 24	IMC 228650
25	JK 25	IMC 228651
26	JK 26	IMC 228652
27	JK 27	IMC 228653
28	JK 28	IMC 228654
29	JK 29	IMC 228655
30	JK 30	IMC 228656
31	JK 31	IMC 228657
32	JK 32	IMC 228658
33	JK 33	IMC 228659
34	JK 34	IMC 228660
35	JK 35	IMC 228661
36	JK 36	IMC 228662
37	JK 37	IMC 228663
38	JK 38	IMC 228664
39	JK 39	IMC 228665
40	JK 40	IMC 228666
41	JK 41	IMC 228667
42	JK 42	IMC 228668

Claim #	Claim Name	BLM #
43	JK 43	IMC 228669
44	JK 44	IMC 228670
45	JK 45	IMC 228671
46	JK 46	IMC 228672
47	JK 47	IMC 228673
48	JK 48	IMC 228674
49	JK 49	IMC 228675
50	JK 50	IMC 228676
51	JK 51	IMC 228677
52	JK 52	IMC 228678
53	JK 53	IMC 228679
54	JK 54	IMC 228680
55	JK 55	IMC 228681
56	JK 56	IMC 228682
57	JK 57	IMC 228683
58	JK 58	IMC 228684
59	JK 59	IMC 228685
60	JK 60	IMC 228686
61	JK 61	IMC 228687
62	JK 62	IMC 228688
63	JK 63	IMC 228689
64	JK 64	IMC 228690
65	JK 65	IMC 228691
66	JK 66	IMC 228692
67	JK 67	IMC 228693
68	JK 68	IMC 228694
69	JK 69	IMC 228695
70	JK 70	IMC 228696
71	JK 71	IMC 228697
72	JK 72	IMC 228698
73	JK 73	IMC 228699
74	JK 74	IMC 228700
75	JK 75	IMC 228701
76	JK 76	IMC 228702
77	JK 77	IMC 228703
78	JK 78	IMC 228704
79	JK 79	IMC 228705
80	JK 80	IMC 228706
81	JK 81	IMC 228707
82	JK 82	IMC 228708
83	JK 83	IMC 228709
84	JK 84	IMC 228710
85	JK 85	IMC 228711
86	JK 86	IMC 228712

Claim #	Claim Name	BLM #
87	JK 87	IMC 228713
88	JK 88	IMC 228714
89	JK 89	IMC 228715
90	JK 90	IMC 228716
91	JK 91	IMC 228717
92	JK 92	IMC 228718
93	JK 93	IMC 228719
94	JK 94	IMC 228720
95	JK 95	IMC 228721
96	JK 96	IMC 228722
97	JK 97	IMC 228723
98	JK 98	IMC 228724
99	JK 99	IMC 228725
100	JK 100	IMC 228726
101	JK 101	IMC 228727
102	JK 102	IMC 228728
103	JK 103	IMC 228729
104	JK 104	IMC 228730
105	JK 105	IMC 228731
106	JK 106	IMC 228732
107	JK 107	IMC 228733
108	JK 108	IMC 228734
109	JK 109	IMC 228735
110	JK 110	IMC 228736
111	JK 111	IMC 228737
112	JK 112	IMC 228738
113	JK 113	IMC 228739
114	JK 114	IMC 228740
115	JK 115	IMC 228741
116	JK 116	IMC 228742
117	JK 117	IMC 228743
118	JK 118	IMC 228744
119	JK 119	IMC 228745
120	JK 120	IMC 228746
121	JK 121	IMC 228747
122	JK 122	IMC 228748
123	JK 123	IMC 228749
124	JK 124	IMC 228750
125	JK 125	IMC 228751
126	JK 126	IMC 228752
127	JK 127	IMC 228753
128	JK 128	IMC 228754
129	JK 129	IMC 228755
130	JK 130	IMC 228756

Claim #	Claim Name	BLM #
131	JK 131	IMC 228757
132	JK 132	IMC 228758
133	JK 133	IMC 228759
134	JK 134	IMC 228760
135	JK 135	IMC 228761
136	JK 136	IMC 228762
137	JK 137	IMC 228763
138	JK 138	IMC 228764
139	JK 139	IMC 228765
140	JK 140	IMC 228766
141	JK 141	IMC 228767
142	JK 142	IMC 228768
143	JK 143	IMC 228769
144	JK 144	IMC 228770
145	JK 145	IMC 228771
146	JK 146	IMC 228772
147	JK 147	IMC 228773
148	JK 148	IMC 228774
149	JK 149	IMC 228775
150	JK 150	IMC 228776
151	JK 151	IMC 228777
152	JK 152	IMC 228778
153	JK 153	IMC 228779
154	JK 154	IMC 228780
155	JK 155	IMC 228781
156	JK 156	IMC 228782
157	JK 157	IMC 228783
158	JK 158	IMC 228784
159	JK 159	IMC 228785
160	JK 160	IMC 228786
161	JK 161	IMC 228787
162	JK 162	IMC 228788
163	JK 163	IMC 228789
164	JK 164	IMC 228790
165	JK 165	IMC 228791

Part 5: 73 Unpatented Lode Claims Owned or Controlled by DeLamar Mining Co.

Claim #	Claim Name	BLM #
1	DS 1	IMC-228903
2	DS 2	IMC-228904
3	DS 3	IMC-228905
4	DS 4	IMC-228906
5	DS 5	IMC-228907
6	DS 6	IMC-228908
7	DS 7	IMC-228909
8	DS 8	IMC-228910
9	DS 9	IMC-228911
10	DS 10	IMC-228912
11	DS 11	IMC-228913
12	DS 12	IMC-228914
13	DS 13	IMC-228915
14	DS 14	IMC-228916
15	DS 15	IMC-228917
16	DS 16	IMC-228918
17	DS 17	IMC-228919
18	DS 18	IMC-228920
19	DS 19	IMC-228921
20	DS 20	IMC-228922
21	DS 21	IMC-228923
22	DS 22	IMC-228924
23	DS 23	IMC-228925
24	DS 24	IMC-228926
25	DS 25	IMC-228927
26	DS 26	IMC-228928
27	DS 27	IMC-228929
28	DS 28	IMC-228930
29	DS 29	IMC-228931
30	DS 30	IMC-228932
31	DS 31	IMC-228933
32	DS 32	IMC-228934
33	DS 33	IMC-228935
34	DS 34	IMC-228936
35	DS 35	IMC-228937
36	DS 36	IMC-228938
37	DS 37	IMC-228939
38	DS 38	IMC-228940
39	DS 39	IMC-228941
40	DS 40	IMC-228942
41	DS 41	IMC-228943
42	DS 42	IMC-228944

Claim #	Claim Name	BLM #
43	DS 43	IMC-228945
44	DS 44	IMC-228946
45	DS 45	IMC-228947
46	DS 46	IMC-228948
47	DS 47	IMC-228949
48	DS 48	IMC-228950
49	DS 49	IMC-228951
50	DS 50	IMC-228952
51	DS 51	IMC-228953
52	DS 52	IMC-228954
53	DS 53	IMC-228955
54	DS 54	IMC-228956
55	DS 55	IMC-228957
56	DS 56	IMC-228958
57	DS 57	IMC-228959
58	DS 58	IMC-228960
59	DS 59	IMC-228961
60	DS 60	IMC-228962
61	DS 61	IMC-228963
62	DS 62	IMC-228964
63	DS 63	IMC-228965
64	DS 64	IMC-228966
65	DS 65	IMC-228967
66	DS 66	IMC-228968
67	DS 67	IMC-228969
68	DS 68	IMC-228970
69	DS 69	IMC-228971
70	DS 70	IMC-228972
71	DS 71	IMC-228973
72	DS 72	IMC-228974
73	DS 73	IMC-228975

APPENDIX B

METALLURGICAL TEST RESULTS

DELAMAR AREA

Table DLM1. - Drill Hole Composite Summary, DeLamar 2018/2019 Testing

Area	Drill Hole	Interval (m)[1]		Composites			
		from	to	Oxidized	Transitional	UnOxidized	Mixed[2]
North DeLamar	IDM18_025	28.596	33.528	0	1	0	0
North DeLamar	IDM18_039	55.169	74.676	0	0	1	0
North DeLamar	IDM18_017	19.507	27.127	0	1	0	0
North DeLamar	IDM18_027	15.24	60.655	0	1	0	1
North DeLamar	IDM18_028	23.317	57.607	1	1	1	0
Glen Silver	IDM18_009	18.288	164.592	3	0	2	2
Glen Silver	IDM18_013[3]	54.559	72.131	0	0	1	0
Glen Silver	IDM18_023	41.453	127.559	1	0	4	0
Glen Silver	IDM18_030	96.926	155.875	0	0	1	0
Sommercamp	IDM18_029	19.812	213.36	1	0	5	0
Sullivan Gulch	IDM18_005	92.964	274.32	0	1	8	2
Sullivan Gulch	IDM18_007	185.928	335.28	0	0	8	0
Sullivan Gulch	IDM18_008	71.628	265.176	0	0	5	0
Sullivan Gulch	IDM18_011	71.628	266.7	0	0	8	0
Sullivan Gulch	IDM18_012	301.752	359.664	0	0	2	0
Sullivan Gulch	IDM18_014	166.116	396.24	0	0	13	0
Sullivan Gulch	IDM18_046	275.844	381	0	0	8	0
Sullivan Gulch	IDM18_047	263.652	379.476	0	0	4	0
Sullivan Gulch	IDM18_048	115.824	428.244	0	0	11	0
Sullivan Gulch	IDM18_052	206.959	342.748	0	0	1	0
Sullivan Gulch	IDM18_055	190.043	204.216	0	0	1	0
Sullivan Gulch North	IDM18_053	109.423	118.567	0	0	1	0
Sullivan Gulch North	IDM18_054	80.315	88.087	0	0	1	0
DeLamar Deposit Total				**6**	**5**	**86**	**5**

1) Not all core within range was used for composites. Samples were composited based on oxidation, lithology, alteration, grade and continuity.

2) Composite contains more material from multiple oxidation classes.

3) Contains drill core from both holes IDM18_13 and IDM18_30.

Table DLM2. - Summary Bottle Roll Test Results, 80%-1.7mm Feed Size,
96 Hour Leach (no interim sampling) at 40% Solids and 1.0 g NaCN/L, DeLamar 2018-2019 Samples

Sample Description						Au Recovery (%)	Head Grade (g Au/t)	Ag Recovery (%)	Head Grade (g Ag/t)	Reagent Requirements (kg/t)	
Composite	Type	Oxidation	Drill Hole	Interval (m) from	Interval (m) to					NaCN Conc.	Lime Added
DeLamar Area											
4307-B	Bulk Sample	ox				75.0	0.24	40.0	5	0.11	4.2
4307-162	Core	ox	IDM18_028	23.47	29.718	58.1	0.31	41.7	36	0.22	2.3
4307-163	Core	ox	IDM18_028	34.138	39.3	27.5	0.40	45.7	35	0.22	2.3
4307-A	Bulk Sample	trans	N/A			66.4	1.10	53.3	15	0.48	7.4
4307-C	Bulk Sample	trans	N/A			81.0	0.42	43.3	30	0.14	3.4
4307-D	Bulk Sample	trans	N/A			56.5	0.62	30.0	10	0.17	3.7
4307-161	Core	trans	IDM18_017	19.507	27.1	83.3	0.18	55.7	70	0.15	2.1
4307-144	Core	unox	IDM18_025	28.651	33.5	48.4	0.31	36.8	38	0.15	2.0
4307-067	AR	unox	IDM18_039	55.169	74.7	4.7	0.43	18.9	53	0.07	1.5
North DeLamar											
4307-059	AR	trans	IDM18_027	15.24	31.2	13.6	0.44	42.9	49	0.30	2.0
4307-060	AR	mixed (trans/unox)	IDM18_027	35.966	60.7	20.0	0.40	45.6	57	0.45	2.8
Glen Silver											
4307-048	AR	ox	IDM18_009	18.288	38.1	72.9	0.48	45.5	11	0.23	5.0
4307-049	AR	ox	IDM18_009	53.34	62.5	83.7	0.43	35.7	14	0.23	2.0
4307-050	AR	ox	IDM18_009	64.008	82.3	90.5	0.63	37.5	8	0.15	2.0
4307-055	AR	ox	IDM18_023	41.453	52.4	80.8	0.52	42.3	26	0.22	2.0
4307-051	AR	mixed (ox/trans)	IDM18_009	89.916	97.5	75.0	0.52	33.3	6	0.15	2.0
4307-052	AR	mixed (trans/unox)	IDM18_009	97.536	108.2	25.6	0.43	20.0	5	0.37	5.1
4307-053	AR	unox	IDM18_009	111.252	121.9	14.0	0.43	16.7	6	0.30	5.9
4307-054	AR	unox	IDM18_009	146.304	164.6	13.2	0.38	20.0	5	0.30	2.6
4307-056	AR	unox	IDM18_023	66.446	85.8	7.4	0.68	14.3	7	0.30	5.0
4307-057	AR	unox	IDM18_023	87.782	116.7	6.6	1.22	10.0	10	0.45	2.7
4307-058	AR	unox	IDM18_023	116.738	127.6	7.8	1.41	18.2	11	0.45	2.0
Sommercamp											
4307-061	AR	ox	IDM18_029	19.812	29.0	80.0	0.30	42.1	19	0.15	5.0
4307-062	AR	unox	IDM18_029	118.872	131.1	7.0	0.43	14.3	4	0.30	1.5
4307-063	AR	unox	IDM18_029	135.636	147.8	11.3	1.59	34.1	44	0.30	2.0
4307-064	AR	unox	IDM18_029	152.4	172.2	17.8	0.45	31.3	16	0.45	2.0
4307-065	AR	unox	IDM18_029	172.212	189.0	19.3	0.57	25.0	8	0.38	2.0
4307-066	AR	unox	IDM18_029	196.596	213.4	13.5	0.37	35.7	14	0.08	1.8

Note: AR denotes assay reject composites. Core denotes split drill core composites. Ox denotes oxidized; trans denotes transitional; unox denotes unoxidized.

Table DLM3. - Bottle Roll Tests, 80%-1.7mm Feed Size,
96 Hour Leach Time (no interim sampling), 40% Solids, 1.0 g NaCN/L, Sullivan Gulch 2018-2019 Composites

Sample Description						Au Recovery	Head Grade	Ag Recovery	Head Grade	Reagent Requirements (kg/t)	
Composite	Type	Oxidation	Drill Hole	Interval (m) from	Interval (m) to	(%)	(g Au/t)	(%)	(g Ag/t)	NaCN Conc.	Lime Added
4307-001	AR	mixed (unox/ox)	IDM18_005	92.964	106.68	47.6	0.42	50.0	10	0.30	2.3
4307-002	AR	mixed (ox/trans)	IDM18_005	106.68	121.92	55.3	0.85	30.0	20	0.15	1.0
4307-003	AR	trans	IDM18_005	121.92	129.54	76.7	0.43	50.0	16	<0.07	1.5
4307-004	AR	unox	IDM18_005	129.54	152.4	49.3	0.71	28.3	166	0.38	2.5
4307-005	AR	unox	IDM18_005	152.4	167.64	16.2	1.11	37.8	45	0.15	1.0
4307-006	AR	unox	IDM18_005	167.64	182.88	21.2	0.66	40.0	50	0.15	1.0
4307-007	AR	unox	IDM18_005	182.88	198.12	20.0	0.45	38.5	26	0.15	1.1
4307-008	AR	unox	IDM18_005	198.12	213.36	19.4	0.31	42.1	19	0.15	1.3
4307-009	AR	unox	IDM18_005	213.36	228.6	23.7	0.38	37.5	24	0.16	1.0
4307-010	AR	unox	IDM18_005	228.6	243.84	37.5	0.32	43.8	32	0.15	1.5
4307-011	AR	unox	IDM18_005	248.412	274.32	42.4	0.33	27.3	11	0.08	1.2
4307-012	AR	unox	IDM18_007	185.928	196.596	12.4	2.01	37.3	59	0.37	2.0
4307-013	AR	unox	IDM18_007	196.596	213.36	27.3	0.88	26.5	83	0.15	2.7
4307-014	AR	unox	IDM18_007	213.36	236.22	26.2	0.42	28.9	45	<0.07	4.3
4307-015	AR	unox	IDM18_007	236.22	259.08	33.3	0.33	50.0	30	0.45	3.9
4307-016	AR	unox	IDM18_007	259.08	281.94	46.4	0.69	35.6	45	0.30	2.7
4307-017	AR	unox	IDM18_007	281.94	301.752	26.2	0.42	26.3	57	0.15	1.9
4307-018	AR	unox	IDM18_007	301.752	320	27.3	0.33	37.1	35	0.22	1.7
4307-019	AR	unox	IDM18_007	320	335	28.6	0.63	38.5	39	0.30	1.8
4307-020	AR	unox	IDM18_008	71.628	86.868	30.0	0.30	42.9	7	0.15	2.4
4307-021	AR	unox	IDM18_008	97.536	112.776	30.4	0.46	48.6	37	0.15	2.5
4307-022	AR	unox	IDM18_008	112.776	128.016	38.6	0.44	38.1	21	0.30	2.9
4307-023	AR	unox	IDM18_008	128.016	143.256	27.5	0.40	27.3	326	0.30	3.2
4307-024	AR	unox	IDM18_008	257.556	265.176	0.0	0.30	33.3	12	0.08	1.0
4307-025	AR	unox	IDM18_011	71.628	83.82	48.6	3.13	32.1	442	0.38	1.0
4307-026	AR	unox	IDM18_011	83.82	99.06	20.5	0.44	33.3	51	0.15	1.0
4307-027	AR	unox	IDM18_011	99.06	114.3	15.8	0.76	27.3	143	0.30	1.0
4307-028	AR	unox	IDM18_011	114.3	135.636	31.1	0.45	39.5	43	0.30	1.5
4307-029	AR	unox	IDM18_011	135.636	147.828	38.5	2.34	31.2	452	0.75	3.3
4307-030	AR	unox	IDM18_011	153.924	179.832	40.5	0.37	39.3	28	0.15	1.8
4307-031	AR	unox	IDM18_011	185.928	222.504	21.1	0.38	40.6	32	0.15	1.0
4307-032	AR	unox	IDM18_011	233.172	266.7	17.4	0.46	35.1	37	0.23	1.0
4307-033	AR	unox	IDM18_012	301.752	323	40.3	0.67	32.4	37	0.15	1.0
4307-034	AR	unox	IDM18_012	337	360	68.9	0.61	50.0	8	0.15	1.0
4307-035	AR	unox	IDM18_014	166.116	182.88	18.8	0.32	44.4	54	0.30	2.9
4307-036	AR	unox	IDM18_014	182.88	204.216	25.3	0.95	31.3	64	0.38	5.0
4307-037	AR	unox	IDM18_014	204.216	210.312	83.4	6.32	44.7	226	0.37	8.1
4307-038	AR	unox	IDM18_014	210.312	228.6	49.7	1.57	40.7	150	0.23	7.4
4307-039	AR	unox	IDM18_014	228.6	243.84	45.7	2.47	37.0	73	0.67	5.4
4307-040	AR	unox	IDM18_014	243.84	259.08	37.4	2.06	34.2	202	0.07	3.3

Note: AR denotes assay reject composites. Core denotes split drill core composites. Ox denotes oxidized; trans denotes transitional; unox denotes unoxidized.

Table DLM4. - Whole Ore Milling/Cyanidation (Bottle Roll) Tests, DeLamar 2018-2019 Drll Core Composites, 80%-75µm Feed Size, 72 Hour Leach Time (with interim sampling and reagent maintenance), 40% Solids, 1.0 g NaCN/L

Sample Description							Au Recovery	Head Grade	Ag Recovery	Head Grade	Reagent Requirements (kg/t)	
Composite	Zone	Type	Oxidation	Drill Hole	Interval (m) from	Interval (m) to	(%)	(g Au/t)	(%)	(g Ag/t)	NaCN Conc.	Lime Added
4307-162	North DeLamar	Core	ox	IDM18_028	23.317	29.718	61.8	0.34	80.0	35	1.54	2.8
4307-144	North DeLamar	Core	trans	IDM18_025	28.596	33.528	64.3	0.42	61.1	36	0.62	3.4
4307-161	North DeLamar	Core	trans	IDM18_017	19.507	27.127	88.9	0.18	84.4	77	1.00	2.7
4307-163	North DeLamar	Core	trans	IDM18_028	34.138	39.319	34.3	0.35	76.5	34	1.78	2.5
4307-164	North DeLamar	Core	unox	IDM18_028	48.463	57.607	59.4	0.64	76.7	30	1.56	4.2
4307-119	Glen Silver	AR	unox	IDM18_023	87.63	127.559	12.5	1.20	22.2	9	0.90	4.3
4307-145/146MC	Glen Silver	Core	unox	IDM18_013/IDM18_030	54.559	72.131	33.3	0.75	50.0	12	0.64	3.0
4307-147/148MC	Glen Silver	Core	unox	IDM18_030	97.231	155.875	15.1	0.53	44.4	9	0.17	2.2
4307-005	Sullivan Gulch	AR	unox	IDM18_005	152.4	167.64	21.3	1.08	52.4	42	0.84	2.3
4307-012	Sullivan Gulch	AR	unox	IDM18_007	185.928	196.596	19.3	1.81	54.5	66	0.80	4.5
4307-025	Sullivan Gulch	AR	unox	IDM18_011	71.628	83.82	59.9	3.09	32.7	505	0.87	2.1
4307-029	Sullivan Gulch	AR	unox	IDM18_011	135.636	147.828	45.0	2.29	30.6	480	1.49	5.4
4307-046	Sullivan Gulch	AR	unox	IDM18_014	372	396	67.3	1.10	47.4	57	0.70	2.6
4307-047	Sullivan Gulch	AR	unox	IDM18_014	204.216	272.796	70.7	2.46	56.2	153	1.86	5.7
4307-120	Sullivan Gulch	AR	unox	IDM18_046	275.844	325	81.1	5.19	50.0	24	0.17	2.0
4307-121	Sullivan Gulch	AR	unox	IDM18_047	263.652	283.464	63.2	1.71	44.4	153	0.86	2.7
4307-149-153MC	Sullivan Gulch	Core	unox	IDM18_052	206.959	342.7	45.3	1.06	41.3	46	0.24	3.0
4307-154	Sullivan Gulch	Core	unox	IDM18_055	190.043	204.216	47.8	0.69	28.6	28	0.17	2.0
4307-165	Sullivan Gulch North	Core	unox	IDM18_053	109.423	118.567	20.7	0.29	40.0	15	1.50	5.1
4307-166	Sullivan Gulch North	Core	unox	IDM18_054	80.315	88.087	40.0	0.35	29.3	41	0.23	1.2

Note: AR denotes assay reject composites. Core denotes split drill core composites.

Table DLM5. - Summary Results, Whole Ore Gravity Concentration with Flotation of Gravity Rougher Tailings, DeLamar 2018-2019 Composites, 80%-75µm Feed Size

Area	Sullivan Gulch								Glen Silver
Composite	4307-005	4307-012	4307-025	4307-029	4307-046	4307-047	4307-120	4307-121	4307-119
Grav. Test No.	G-1	G-2	G-3	G-4	G-5	G-6	G-8	G-9	G-7
Flot. Test No.	F-7	F-8	F-9	F-10	F-11	F-12	F-14	F-15	F-13
Head Assay (% Sulfide Sulfur)	1.04	2.82	2.01	2.57	8.44	2.66*	2.66*	2.66*	2.66*
Weight (%)									
Gravity Cl. Conc.	0.7	1.3	1.2	1.0	2.1	0.6	1.1	1.4	1.2
Gravity Cl. Tail	2.2	3.2	3.1	2.7	3.5	2.8	1.6	2.0	2.5
Flotation Cl. Conc.	7.9	4.3	6.0	10.2	11.5	12.4	3.3	4.7	4.0
Combined (Grav.+ Flot. Cl. Conc.)	10.1	7.5	9.1	12.9	15.0	15.2	4.9	6.7	6.5
Flotation Cl. Tail	15.3	14.2	33.7	9.7	29.9	10.0	5.5	6.7	19.4
Combined (Grav.+ Flot. Ro. Conc.)	25.4	21.7	42.8	22.6	44.9	25.2	10.4	13.4	25.9
Flotation Ro. Tail	73.9	77.0	56.0	76.4	53.0	74.2	88.5	85.3	72.9
Grade (g Au/t)									
Gravity Cl. Conc.	18.2	23.9	95.4	32.4	19.5	94.8	185.0	54.2	16.0
Gravity Cl. Tail	2.43	5.68	3.87	3.39	5.28	11.40	19.50	3.37	4.50
Flotation Cl. Conc.	6.07	10.80	9.77	11.60	3.37	10.70	46.40	9.38	6.91
Combined (Grav.+ Flot. Cl. Conc.)	6.54	12.76	20.34	12.39	6.55	14.57	79.15	18.91	8.94
Flotation Cl. Tail	1.68	2.83	2.02	3.64	0.54	1.49	19.30	1.37	1.97
Combined (Grav.+ Flot. Ro. Conc.)	3.61	6.26	5.92	8.64	2.55	9.38	47.50	10.14	3.72
Flotation Ro. Tail	0.20	0.57	0.72	0.29	0.15	0.11	0.17	0.23	0.46
Calculated Head	1.07	1.80	2.93	2.17	1.22	2.45	5.09	1.56	1.30
Grade (g Ag/t)									
Gravity Cl. Conc.	610	730	9,470	5,950	447	2,810	483	5,820	104
Gravity Cl. Tail	107	174	723	444	266	534	77	393	31
Flotation Cl. Conc.	229	420	2,610	3,230	172	796	320	1,530	61
Combined (Grav.+ Flot. Cl. Conc.)	245	442	3,216	3,108	257	859	349	2,407	69
Flotation Cl. Tail	60	86	426	541	40	120	104	143	17
Combined (Grav.+ Flot. Ro. Conc.)	133	209	1,019	2,006	112	566	219	1,275	30
Flotation Ro. Tail	9	16	100	14	9	10	1	12	3
Calculated Head	41	58	492	464	55	150	24	181	10
Au Distribution (% of total)									
Gravity Cl. Conc.	12.0	17.3	39.0	14.9	33.5	23.3	40.0	48.8	14.8
Gravity Cl. Tail	5.0	10.1	4.1	4.2	15.1	13.1	6.1	4.3	8.7
Flotation Cl. Conc.	45.0	25.8	20.0	54.4	31.7	54.2	30.1	28.4	21.3
Combined (Grav.+ Flot. Cl. Conc.)	62.0	53.2	63.1	73.5	80.3	90.6	76.2	81.5	44.8
Flotation Cl. Tail	24.1	22.4	23.2	16.3	13.2	6.1	20.9	5.9	29.4
Combined (Grav.+ Flot. Ro. Conc.)	86.1	75.6	86.3	89.8	93.5	96.7	97.1	87.4	74.2
Flotation Ro. Tail	13.9	24.4	13.7	10.2	6.5	3.3	3.0	12.6	25.8
Ag Distribution (% of total)									
Gravity Cl. Conc.	10.5	16.5	23.1	12.8	17.0	11.2	22.4	45.0	12.5
Gravity Cl. Tail	5.8	9.7	4.6	2.6	16.9	10.0	5.2	4.3	7.8
Flotation Cl. Conc.	44.6	31.3	31.8	71.0	35.8	65.8	44.5	39.7	24.5
Combined (Grav.+ Flot. Cl. Conc.)	60.9	57.5	59.5	86.4	69.7	87.0	72.1	89.0	44.8
Flotation Cl. Tail	22.7	21.2	29.1	11.3	21.7	8.0	24.1	5.3	33.2
Combined (Grav.+ Flot. Ro. Conc.)	83.6	78.7	88.6	97.7	91.4	95.0	96.2	94.3	78.0
Flotation Ro. Tail	16.4	21.3	11.4	2.3	8.6	5.0	3.7	5.7	22.0

* Predicted total sulfur head grade.

Table DLM6. - Summary Flotation Test Results, Optimization Testing, 2018-2019 McClelland Testing

	Flotation Cleaner Concentrate					Flotation Rougher Concentrate							
	Mass	Grade		Recovery		Mass	Grade		Recovery		Head Grade		
Feed Size	(%)	(g Au/t)	(g Ag/t)	(% Au)	(% Ag)	(%)	(g Au/t)	(g Ag/t)	(% Au)	(% Ag)	(g Au/t)	(g Ag/t)	(% S=)
4307-149-153MC; Sullivan Gulch; IDM18_052; 679' - 1,124'; unox; Tql lithology; mixed alteration													
80%-212µm	9.8	8.8	377	82.2	80.1	15.5	6.20	287	91.9	96.3	1.05	46	2.17
80%-150µm	4.1	18.2	826	73.4	82.5	12.6	7.51	312	93.1	95.7	1.02	41	2.21
80%-75µm	7.7	9.8	545	69.4	90.2	17.8	5.70	252	93.2	96.5	1.09	47	2.28
4307-154; Sullivan Gulch; IDM18_055; 623.5' - 670'; unox; Tpl lithology; mixed alteration													
80%-212µm	7.7	10.7	249	80.8	86.4	19.5	4.69	106	89.7	92.8	1.02	22	3.76
80%-150µm	8.2	8.2	255	87.3	86.0	21.3	3.42	110	94.8	96.8	0.77	24	3.85
80%-75µm	11.4	5.0	206	85.6	92.5	25.9	2.42	95	94.4	97.1	0.66	25	3.31
4307-145/146MC; Glen Silver; IDM18_013/IDM18_030; 179' - 236.65'; unox; Tpr lithology; mixed alteration													
80%-75µm	5.0	7.2	128	48.6	54.1	19.7	2.77	44	73.9	72.8	0.74	12	1.26
80%-45µm	4.4	10.2	151	57.9	60.0	9.6	5.82	87	72.0	75.5	0.78	11	1.32
4307-147/148MC; Glen Silver; IDM18_030; 319' - 511.4'; unox; Tql lithology; mixed alteration													
80%-75µm	6.1	5.1	99	60.7	79.5	13.8	2.83	49	76.4	88.6	0.51	8	1.33
80%-45µm	3.6	6.1	148	45.7	66.5	8.8	3.74	81	67.9	88.6	0.48	8	1.23

FLORIDA MOUNTAIN AREA

Table FM1. - Drill Hole Composite Summary, Florida Mountain 2018-2019 Testing

Area	Drill Hole	Interval (m)[1]		Composites			
		from	to	Oxidized	Transitional	UnOxidized	Mixed[2]
Florida Mountain	IFM18_001	5.639	16.916	0	1	0	0
Florida Mountain	IFM_18_001A	11.735	313.334	0	7	2	1
Florida Mountain	IFM18_003	0	161.849	2	2	6	1
Florida Mountain	IFM18_004	122.53	191.414	0	0	5	0
Florida Mountain	IFM18_010	17.678	169.469	0	7	4	0
Florida Mountain	IFM18_012	4.877	119.786	0	2	1	0
Florida Mountain	IFM18_025	10.058	39.014	0	3	0	0
Florida Mountain	IFM18_026A	14.478	111.862	0	2	4	0
Florida Mountain	IFM_18_003[3]	122.53	169.469	0	0	1	0
Total				**2**	**24**	**23**	**2**

1) Not all core within range was used for composites. Samples were composited based on oxidation, lithology, alteration, grade and continuity.

2) Composite contains more material from multiple oxidation classes.

3) Contains drill core from holes IFM18_003, IFM18_004 and IFM18_010.

Table FM2. - Summary Bottle Roll Test Results, 80%-1.7mm Feed Size, 96 Hour Leach Time at 40% Solids and 1.0 g NaCN/L, Florida Mountain 2018-2019 Composites

Sample Description						Au Recovery	Head Grade	Ag Recovery	Head Grade	Reagent Requirements (kg/t)	
Drill				Interval (m)							
Composite	Type	Oxidation	Hole	from	to	(%)	(g Au/t)	(%)	(g Ag/t)	NaCN Conc.	Lime Added
4307-126	Core	ox	IFM18_003	0	32.918	83.6	0.73	37.5	8	0.16	1.0
4307-100	AR	ox	IFM18_003	3.962	32.918	74.5	0.47	66.7	6	0.08	1.0
4307-096	AR	trans	IFM18_001	5.639	16.916	82.7	1.27	44.0	25	0.15	1.4
4307-122	Core	trans	IFM18_001A	11.735	20.269	89.4	0.47	62.5	16	0.24	1.1
4307-097	AR	trans	IFM18_001A	39.014	56.388	69.1	0.68	53.7	67	0.23	1.3
4307-123	Core	trans	IFM18_001A	39.014	56.388	83.9	0.31	58.8	80	0.21	1.3
4307-098	AR	trans	IFM18_001A	66.446	87.478	40.6	0.96	33.3	3	0.07	1.0
4307-124	Core	trans	IFM18_001A	66.446	81.686	60.0	0.40	25.0	4	0.08	0.8
4307-101	AR	trans	IFM18_003	43.586	71.171	85.7	0.70	35.3	17	0.15	1.2
4307-127	Core	trans	IFM18_003	43.586	71.171	89.7	0.78	27.3	22	0.24	1.2
4307-107	AR	trans	IFM18_010	17.678	32.918	88.0	0.50	53.3	15	0.00	0.8
4307-129	Core	trans	IFM18_010	17.678	32.918	73.9	0.46	50.0	14	0.19	0.7
4307-108	AR	trans	IFM18_010	32.918	51.206	91.1	0.45	46.2	13	0.00	0.8
4307-130	Core	trans	IFM18_010	32.918	51.206	75.5	0.53	47.7	44	0.14	1.8
4307-109	AR	trans	IFM18_010	51.206	69.494	94.6	0.92	50.0	14	0.14	1.1
4307-131	Core	trans	IFM18_010	51.206	69.494	85.6	1.04	41.7	12	0.18	0.9
4307-132	Core	trans	IFM18_012	4.877	29.87	86.0	0.86	47.8	136	0.28	1.1
4307-112	AR	trans	IFM18_012	10.363	29.87	93.0	1.15	38.9	190	0.29	1.2
4307-114	AR	trans	IFM18_025	10.058	26.822	89.4	0.66	33.3	9	0.15	2.5
4307-133	Core	trans	IFM18_025	10.058	39.014	80.9	0.47	37.5	16	0.11	0.8
4307-115	AR	trans	IFM18_025	26.822	39.014	86.9	0.61	54.3	116	0.67	1.2
4307-116	AR	trans	IFM18_026A	14.478	29.413	87.2	1.95	57.3	124	0.29	1.3
4307-134	Core	trans	IFM18_026A	14.478	29.413	85.3	2.31	62.0	79	0.10	1.3
4307-099	AR	unox	IFM18_001A	291.998	313	72.1	1.54	25.7	249	0.15	1.0
4307-125	Core	unox	IFM18_001A	291.998	313	84.3	0.70	19.0	274	0.28	0.4
4307-102	AR	unox	IFM18_003	109.118	119.482	19.0	0.42	25.0	4	0.07	2.1
4307-103	AR	unox	IFM18_003	122.834	147.371	39.1	0.46	20.0	5	0.22	1.6
4307-128	Core	unox	IFM18_003	122.834	131.978	13.9	0.36	25.0	4	0.31	1.0
4307-135	Core	unox	IFM18_003	122.834	161.849	35.9	0.64	20.0	10	0.15	2.0
4307-104	AR	unox	IFM18_003	147.371	161.849	39.4	0.66	25.0	8	0.45	1.7
4307-105	AR	unox	IFM18_004	122.53	131.674	52.7	1.12	36.4	11	0.07	2.5
4307-106	AR	unox	IFM18_004	182.27	191.414	34.4	0.32	42.9	7	0.15	2.5
4307-110	AR	unox	IFM18_010	99.974	113.69	46.2	0.39	50.0	2	0.37	1.3
4307-111	AR	unox	IFM18_010	131.064	149.352	60.5	0.43	66.7	3	0.22	1.0
4307-113	AR	unox	IFM18_012	110.642	119.786	74.5	0.47	33.3	12	0.74	2.2
4307-117	AR	unox	IFM18_026A	89.002	95.555	50.0	0.46	50.0	4	0.29	1.6
4307-118	AR	unox	IFM18_026A	106.07	111.862	54.0	0.50	41.2	51	0.59	3.9

Note: AR denotes assay reject composites. Tests on AR composites were conducted without interim sampling and reagent make-up. Core denotes split drill core composites. Tests on split core composites were conducted with interim sampling and reagent maintenance. Ox denotes oxide; trans denotes transitional; unox denotes unoxidized.

Table FM3. - Summary Metallurgical Results, Column Leach Tests, Florida Mountain Drill Core Composites, 80%-12.5mm Feed Size

Sample Description				Leach	Au	Head	Ag	Head	Reagent Requirements (kg/t)	
Composite	Drill Hole	Depth (ft)	Oxidation Class	Time Days	Recovery (%)	Grade (g Au/t)	Recovery (%)	Grade (g Ag/t)	NaCN Conc.	Lime Added
4307-138	IFM18_003	0-108', 143-233.5'	mixed (ox/trans)	65	94.7	0.75	37.5	16	1.16	1.0
4307-132	IFM18_012	16-98'	trans	63	91.3	0.92	43.3	67	1.29	1.0
4307-133	IFM18_025	33-128'	trans	97	85.5	0.69	39.0	59	3.08	0.7
4307-136	IFM18_001A	38.5-66.5', 128-185'	trans	63	87.2	0.39	41.3	75	1.17	1.1
4307-139	IFM18_010	58-228'	trans	65	90.2	0.61	26.3	19	1.18	1.0
4307-137	IFM18_001A	218-268', 958-1028'	mixed (trans/unox)	97	65.7	1.02	30.0	170	2.03	0.5
4307-135	IFM18_003	403-531'	unox	64	30.0	0.60	10.0	10	1.22	1.8

Note: Ox denotes oxide, trans denotes transitional and unox denotes unoxidized.

Table FM4. - "Whole Ore" Milling/Cyanidation Tests, Florida Mountain 2018-2019 Drill Core Composites, 80%-75µm Feed Size, 72 Hour Leach (with interim sampling), 40% Solids, 1.0 g NaCN/L

Sample Description						Au	Head	Ag	Head	Reagent Requirements (kg/t)	
				Interva (m)		Recovery	Grade	Recovery	Grade	NaCN	Lime
Drill Composite	Type	Oxidation	Hole	from	to	(%)	(g Au/t)	(%)	(g Ag/t)	Conc.	Added
4307-155	AR	trans	IFM18_001A	39.014	87.478*	94.4	0.54	92.5	29	0.17	1.7
4307-099	AR	unox	IFM18_001A	291.998	313.334	96.1	1.54	32.7	275	0.20	1.3
4307-135	Core	unox	IFM18_003	122.834	161.849	81.0	0.58	52.8	11	0.16	1.9
4307-156	AR	unox	IFM18_003	109.118	161.849*	76.8	0.56	53.6	6	0.43	2.0
4307-140	Core	unox	IFM18_004	122.53	131.674	87.8	0.98	48.3	6	0.18	3.1
4307-141	Core	unox	IFM18_004	147.218	188.366	81.6	0.49	61.5	8	0.08	2.7
4307-157	AR	unox	IFM18_004	122.530	191.414*	79.7	0.59	66.3	10	0.48	2.5
4307-142	Core	unox	IFM18_010	131.216	169.469	89.8	0.59	63.0	5	0.22	2.2
4307-158	AR	unox	IFM18_010	99.974	149.352*	89.5	0.38	>66.7	<3	<0.07	1.8
4307-159	AR	unox	IFM18_026A	89.002	111.862*	81.8	0.33	79.2	24	0.38	3.5
4307-143	Core	unox	IFM18_026A	89.002	108.814	93.0	0.57	90.8	51	0.31	2.6

* Non-continuous intervals.

Note: AR denotes assay reject composites. Core denotes split drill core composites.

Table FM5. - Gravity Concentration with Agitated Cyanidation of Gravity Tailings, Florida Mountain Comp. 4307-160, 80%-212µm Primary (Gravity Concentration) Grind Size, Reground Gravity Tailings, 72 Hour Leach (with interim sampling), 40% Solids, 1.0 g NaCN/L

	Au Recovery % of total			Head Grade	Ag Recovery % of total			Head Grade	Reagent Requirements (kg/t)	
Regrind Size	Gravity[1]	CN[2]	Combined[3]	(g Au/t)	Gravity[1]	CN[2]	Combined[3]	(g Ag/t)	NaCN Conc.	Lime Added
80%-150µm	7.5	74.3	81.8	0.79	1.8	55.2	57.0	6.9	0.04	1.6
80%-106µm	7.5	74.0	81.5	0.79	1.7	56.2	57.9	7.4	0.10	1.6
80%-75µm	6.8	75.4	82.2	0.87	1.7	58.4	60.1	7.5	0.11	1.6
80%-53µm	7.9	74.2	82.1	0.75	1.6	59.8	61.4	7.7	0.12	1.8
80%-45µm	7.8	73.1	80.9	0.76	1.7	69.6	71.3	7.2	0.02	1.9

1) Recovery to a gravity concentrate produced with a 0.04% mass pull, with a grade of 148 g Au/t and 316 g Ag/t.

2) Recovery by agitated cyanidation of the gravity tailings (99.96% mass).

3) Combined recovery by gravity concentrate and agitated cyanidation of gravity tailings.

Table FM6. - Gravity Concentration with Flotation of Gravity Tailings, Florida Mountain Comp. 4307-160, 80%-212µm Gravity Concentration Feed

	Flotation Cleaner Concentrate[2]					Flotation Rougher Concentrate[2]				
				Recovery		Mass			Recovery	
Regrind Size	Mass, %	(g Au/t)	(g Ag/t)	(% Au)	(% Ag)	(%)	(g Au/t)	(g Ag/t)	(% Au)	(% Ag)
80%-212µm[1]	4.3	17.8	146	96.0	83.8	6.6	11.80	99	97.6	87.5
80%-180µm	3.6	18.1	185	91.2	82.5	8.1	8.31	88	94.9	88.7
80%-150µm	3.2	25.0	180	86.7	75.8	8.2	11.01	83	96.1	88.0
80%-106µm	4.4	17.3	167	92.7	84.3	11.2	6.95	69	95.7	89.8
80%-75µm	2.7	27.7	204	82.4	76.2	10.8	7.51	58	90.1	87.5

1) Gravity concentration feed size was 80%-212µm. No regrind before flotation.

2) Includes gravity cleaner concentrate (0.04% mass pull, 148 g Au/t and 316 g Ag/t), and flotation cleaner concentrate.

Table FM7. - Combined Results, Gravity Concentration/Flotation of Gravity Tailings/Regrind Leach of Flot. Conc., Florida Mountain Unoxidized Master Composite 4307-160, 80%-212µm (65M) Feed Size, 95%-37µm Flotation Concentrate Regrind, 96 Hour Leach, 25% Solids, 5.0 g NaCN/L

		Assay			Distribution			Reagent Requirements (kg/t)	
Product	Weight %	(g Au/t)	(g Ag/t)	(% S=)	(g Au/t)	(g Ag/t)	(% S=)	NaCN Conc.	Lime Added
Gravity Cl. Conc.	0.04	148	316	N/A	8.8	1.5			
Flotation Ro. Conc.	4.58			15.6			84.3		
Recovered, CN		11.81	150		80.9	78.7		0.14	0.2
Leach Tail, CN		0.88	17		6.0	8.9			
Combined Recovery[1]					**89.7**	**80.2**			
Flot. Ro. Tail	95.38	0.03	1	0.14	4.3	10.9	15.7		
Combined Tail	99.96	0.07	2	0.85	10.3	19.8			
Composite	**100.00**	**0.67**	**9**	**0.85**	**100.0**	**100.0**	**100.0**	**0.14**	**0.2**

1) Includes gold and silver reporting to the gravity cleaner concentrate and extracted by cyanidation of the reground flotation concentrate.