MINE DEVELOPMENT ASSOCIATES

A Division of RESPEC

RESPEC

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

Date of Report: October 31, 2023

Effective Date of Report: August 25, 2023

Submitted to:

INTEGRA RESOURCES

1050-400 Burrard Street
Vancouver, BC, Canada V6C 3A6

www.integraresources.com

Qualified Persons:

RESPEC

Thomas L. Dyer, P.E.
Michael M. Gustin, C.P.G.
Jay Nopola, P.E.

McClelland Laboratories, Inc.

Jack S. McPartland, Registered Member, M.M.S.A.

M3 Engineering & Technology Corp.

Matthew Sletten, P.E.
Benjamin Bermudez, P.E.

Welsh Hagen and Associates

John D. Welsh, P.E.

Warm Springs Consulting LLC.

John F. Gardner, P.E.

Elbow Creek Engineering Inc.

Michael M. Botz, P.E.

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

CONTENTS

TABLES

FIGURES

APPENDICES

Frontispiece: aerial view looking easterly from above the DeLamar pit area towards Florida Mountain.

1.0 SUMMARY

RESPEC Company LLC ("RESPEC") has supervised the preparation of this technical report of 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 NYSE American Exchange (NYSE:ITRG). The DeLamar project encompasses the DeLamar and Florida Mountain deposit areas. Both deposit areas have been subject to historical underground mining 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 Thomas L. Dyer, P.E. and Principal Engineer for RESPEC, Michael M. Gustin, C.P.G. and Principal Consultant for RESPEC, Jack McPartland, Registered Member MMSA and Senior Metallurgist with McClelland Laboratories, Inc., John Welsh, P.E., of Welsh Hagen in Reno, Nevada, Matthew Sletten, P.E. and Benjamin Bermudez, P.E. of M3 Engineering & Technology Corp. in Tucson, Arizona, Jay Nopola, P.E, of RESPEC in Rapid City, South Dakota, Michael Botz, P.E., of Elbow Creek Engineering in Billings, Montana, and John F. Gardner, P.E. of Warm Springs Consulting in Boise, Idaho, in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as amended. Mr. Dyer, Mr. Gustin, Mr. McPartland, Mr. Welsh, Mr. Sletten, Mr. Bermudez, Mr. Botz, Mr. Nopola, and Mr. Gardner are Qualified Persons under NI 43-101 and have no affiliation with Integra or its subsidiaries, except that of independent consultant/client relationships.

The effective date of this technical report is August 25, 2023.

This technical report also includes the results of a pre-feasibility study ("PFS") and mineral reserve statement on the DeLamar project included in the NI 43-101 technical report titled "Technical Report and Preliminary Feasibility Study for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated March 22, 2022 with an effective date of January 24, 2022. The results of the PFS and the mineral reserve statement included therein and reproduced in this technical report remain unaffected by the updated mineral resource included herein. The PFS and mineral reserve statement have an effective date of January 24, 2022. Sections 15, 16, 17, 18, 19, 21, 22, 23, and 24 have been reproduced herein and have an effective date of January 24, 2022.

1.1 Property Description and Ownership

The DeLamar project area includes 790 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 8,673 hectares (21,431 acres) in southwestern Idaho, about 80 kilometers (50 miles) 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 $473,244. 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.

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

There are also eight lease agreements covering 33 patented claims and five unpatented claims that require NSR payments ranging from 2.0% 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 and reserves, with 5.0% and 2.5% NSRs applicable to maximums of $50,000 and $650,000 in royalty payments, respectively.

The property includes 1,561 hectares (3,857.2 acres) under seven leases from the State of Idaho, which are subject to a 5.0% net smelter returns production royalty plus annual payments of $27,282. The State of Idaho leases include very small portions of both the DeLamar and Florida Mountain resources and reserves.

Kinross had 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 royalty was subsequently sold to Triple Flag. The Triple Flag royalty applies to more than 90% of the current DeLamar area resources and reserves, but this royalty will be reduced to 1.0% upon Triple Flag receiving total royalty payments of CAD$10,000,000.

On July 28, 2022, the Company executed a credit agreement with Beedie Investments Ltd., for the issuance of a non-revolving term convertible debt facility. The convertible facility is secured by the Company's material assets, including the Delamar Project. In May 2023, Wheaton Precious Metals Corp. acquired from Integra a Right of First Refusal ("ROFR") on all future precious-metals royalties, streams and pre-pays transactions on all properties owned by the Company as of May 4, 2023.

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 DMC lands peripheral to the DeLamar project have no mineral resources or mineral reserves.

The DeLamar project historical open-pit mine areas have been in closure since 2003. While a substantial amount of reclamation and closure work has been completed to date at the site, there remain ongoing water-management activities, monitoring, and reporting. A reclamation bond of $3,276,078 remains with the Idaho Department of Lands ("IDL") and a reclamation bond of $100,000 remains with the Idaho Department of Environmental Quality. Additional reclamation bonds in the total amount of $714,400 have been placed with the U.S. Bureau of Land Management ("BLM") for exploration activities and groundwater well installation on public lands. There are also reclamation bonds with the IDL in the total amount of $155,900 for exploration activities on IDL leased lands.

1.1.1 Exploration and Mining History

Total production of gold and silver from the DeLamar 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 additional but unknown quantity produced at the DeLamar mill in 1999. From 1876 to 1891, an estimated 1.025 million ounces of gold and 51 million ounces of silver were produced from the original De Lamar (as it was historically called) 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 early 1863 in Jordan Creek, a short distance 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, following which very little production took place until 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 tailing 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 (30,000 tons) per day, with a milling capacity of about 3,629 tonnes (4,000 tons) 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.

Exploration of the DeLamar project by Integra commenced in 2017. Since then, Integra has carried out geophysical and geochemical exploration programs, geologic mapping, and exploration, infill, metallurgical, and geotechnical drilling programs. The DeLamar project is an advanced property as defined by NI 43-101.

1.2 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 meters/feet laterally and vertically, and of mainly less than 1.3 centimeters (0.5 inches) 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. The primary 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.

1.3 Drilling, Drill-Hole Database, and Data Verification

As of the effective date of this technical report, the resource database includes data from 3,185 holes, for a total of 372,888 meters (1,223,386 feet), 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 (904,821 feet) 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 (35,581 feet). Approximately 74% of the historical drilling was vertical, including all conventional rotary holes. At DeLamar, a significant portion of the total meterage drilled historically was subsequently mined during the open-pit operations.

Integra commenced drilling in 2018 and has drilled a total of 560 holes (RC, core, and Sonic holes) for a total of 97,098 meters (318,414 feet) in the DeLamar and Florida Mountain areas combined. All but one of the Integra holes were drilled at angles. Integra's drilling continued into 2023, but only the stockpile-related 2023 drilling is included in the resource database used to estimate the current mineral resources; the 2023 drilling campaign is ongoing and consists primarily of geotechnical and metallurgical drilling that will be incorporated into future studies.

The historical portions of the current resource drill-hole databases for the DeLamar and Florida Mountain deposit areas were created by RESPEC using original DeLamar mine digital database files, and this information was subjected to extensive verification measures by both RESPEC and Integra. The Integra portions of the drill-hole databases were directly created by RESPEC using original digital analytical certificates in the case of the assay tables and checking against original digital records in the case of the collar and down-hole deviation tables. Through these and numerous other verification procedures summarized herein, Mr. Gustin has verified that the DeLamar project data as a whole are acceptable as used in this technical report.

1.4 Metallurgical Testing

Metallurgical testing by Integra, generally conducted at McClelland Laboratories ("McClelland") during 2018 through 2023, has been used to select preferred processing methods and estimate recoveries for oxide, mixed and non-oxide mineralization from both the DeLamar and Florida Mountain deposits. Samples used for this testing, primarily drill hole composites from 2018 through 2020 Integra drilling, 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.

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 mixed material types from both deposits can be processed by heap-leach cyanidation. These materials generally benefit from relatively fine crushing to maximize heap-leach recoveries and a feed size of 80% -12.7mm (0.5 inches) was selected as optimum. Expected heap-leach gold recoveries for the oxide mineralization from both deposits (DeLamar and Florida Mountain) are consistently high (70% - 89%). Heap leach gold recoveries for the mixed mineralization are expected to average 72% for Florida

Mountain and to range from 45% to 63% for the DeLamar deposit. Heap leach silver recoveries from the Florida Mountain oxide and mixed materials are expected to average 49% and 47%, respectively. Expected heap-leach silver recoveries from the DeLamar material are highly variable (11% to 74%), but generally low. A significant portion of the DeLamar oxide and mixed mineralization will require agglomeration pretreatment using cement, because of elevated clay content. None of the Florida Mountain heap-leach material is expected to require agglomeration.

Preliminary bottle-roll testing on reverse-circulation and sonic drill intervals from the historic backfill and waste dump materials at DeLamar and Florida Mountain have shown that the materials can be processed by heap-leach cyanidation. Further variability bottle-roll testing and column-leach testing is being completed on the materials to determine ultimate gold and silver recovery estimates. Preliminary heap leach recovery estimates for the historical backfill and waste dump materials were made based on available bottle roll test results and are summarized in Table 1.2.

Metallurgical testing (primarily flotation and agitated cyanidation) has shown that the DeLamar non-oxide materials respond well to flotation at a moderate grind size (150 microns) for recovery of gold and silver to a flotation concentrate. The resulting flotation concentrate responds well to cyanide leaching after very fine regrinding (20 microns) for recovery of contained silver. Some gold is also recovered by cyanide leaching of the reground flotation concentrate, but those recoveries generally are low. Mineralogical examination and metallurgical testing have shown that these materials contain significant amounts of gold that are locked in sulfide mineral particles, which require oxidative pretreatment of sulfide minerals for liberation of gold before high cyanidation gold recoveries can be obtained. Expected recoveries from the DeLamar non-oxide mineralization in the planned mill circuit, consisting of grinding, flotation concentrate regrinding and cyanide leach, range from 28% to 39% for gold and from 64% to 87% for silver.

Metallurgical testing has shown that the non-oxide mineralization from the Florida Mountain deposit responds well to upgrading by flotation at a moderate grind size (150 microns) and cyanidation gold and silver recoveries from the resulting concentrates can be maximized by very fine regrinding (20 microns). In contrast to the DeLamar non-oxide materials, oxidative pretreatment of contained sulfide minerals is not required to achieve high cyanidation gold recoveries from the Florida Mountain non-oxide feeds. Recoveries expected from the Florida Mountain non-oxide mineralization in the planned mill circuit vary with feed grade, but generally are high, with maximum recoveries of 87% gold and 77% silver.

1.5 Mineral Resources

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

- evaluating the drill data statistically and spatially to determine natural gold and silver populations;
- creating low-, medium-, and high-grade mineral-domain polygons for both gold and silver on sets of cross sections spaced at 30-meter (98.4-foot) intervals;
- projecting the sectional mineral-domain polygons horizontally to the drill data within each sectional window;

- slicing the three-dimensionally projected mineral-domain polygons along 6-meter-spaced (19.7-foot) horizontal planes at the DeLamar area and 8-meter-spaced (26.3-foot) planes at Florida Mountain and using these slices to rectify the gold and silver mineral-domain polygons on a set of level plans for each resource area;
- coding a block model to the gold and silver mineral domains for each of the two deposit areas using the level-plan mineral-domain polygons;
- analyzing the modeled mineralization geostatistically to aid in the establishment of estimation and classification parameters; and
- interpolating gold and silver grades by inverse-distance to the third power into 6 x 6 x 6-meter (19.7 x 19.7 x 19.7-foot) blocks for the DeLamar area and 6 x 8 x 8-meter (19.7 x 26.3 x 26.3-foot) blocks at Florida Mountain, using the coded gold and silver mineral-domain percentages to explicitly constrain the grade estimations.

The first-time estimate of stockpile resources, which are comprised of historically mined but not processed materials, were modeled similarly to the in-situ resources, but solids or closely spaced long sections were used instead of level plans.

The DeLamar project mineral resources were estimated to reflect potential open-pit extraction and processing by: crushing and heap leaching of oxide, mixed, and all stockpile materials at both the DeLamar and Florida Mountain areas; grinding, flotation, ultra-fine regrind of concentrates, and Albion cyanide-leach processing of the reground concentrates for the non-oxide materials at the DeLamar area; and grinding, flotation, ultra-fine regrind of concentrates, and agitated cyanide-leaching of non-oxide materials at Florida Mountain. To meet the requirement of having reasonable prospects for eventual economic extraction by open-pit methods, pit optimizations for the DeLamar and Florida Mountain areas were run using the parameters summarized in Table 1-1 and Table 1-2, and the resulting pits were used to constrain the project resources.

Table 1-1 Resource Pit Optimization Cost Parameters

Parameter	DeLamar In Situ	Florida Mtn In Situ	N DM-SC Stockpile	DM #1 + #2 Stockpile	Jacobs Gulch Stockpile	Unit
Mining Cost	$2.00	$2.00	$1.70	$1.70	$1.70	$/tonne mined
Heap Leach						
Oxide Processing	$2.75	$2.75	$5.00	$4.25	$4.00	$/tonne processed
Mixed Processing	$3.75	$3.50	$5.00	$4.25	$4.00	$/tonne processed
Incremental Haulage	$0.20	$0.20	$0.20	$0.20	$0.20	$/tonne processed
G&A Cost	$0.40	$0.40	$0.40	$0.40	$0.40	$/tonne processed
Mill – DeLamar Area						
Non-Oxide Processing	$16.75	$9.75				$/tonne processed
Incremental Haulage	$0.20	$0.20				$/tonne processed
G&A Cost	$0.25	$0.25				$/tonne processed
Au Price	$1,800					$/oz produced
Ag Price	$21.00					$/oz produced
Au Refining Cost	$5.00					$/oz produced
Ag Refining Cost	$0.50					$/oz produced
Royalty	Table 4.2					NSR

Table 1-2 Resource Pit-Optimization Metal Recoveries by Deposit and Oxidation State

	DeLamar In Situ			Florida Mountain In Situ			DeLamar Stockpiles			Florida Mtn Stockpiles
Process Type	Oxide	Mixed	Non-Oxide	Oxide	Mixed	Non-Oxide	N DM-SC	DM #1	DM #2	ALL
Heap Leach – Au	90%	70%	-	90%	75%	-	70%	80%	80%	90%
Heap Leach – Ag	40%	50%	-	55%	60%	-	60%	50%	55%	45%
Mill - Albion - Glen Silver - Au	-	-	78%	-	-	-	-	-	-	-
Mill - Albion - Glen Silver - Ag	-	-	78%	-	-	-	-	-	-	-
Mill - Albion – Milestone - Au	-	-	70%	-	-	-	-	-	-	-
Mill - Albion - Milestone - Ag	-	-	75%	-	-	-	-	-	-	-
Mill - Albion - Other Areas - Au	-	-	87%	-	-	-	-	-	-	-
Mill - Albion - Other Areas - Ag	-	-	87%	-	-	-	-	-	-	-
Mill - Agitated Leach - Au	-	-	-	-	-	95%	-	-	-	-
Mill - Agitated Leach - Au	-	-	-	-	-	92%	-	-	-	-

The pit shells created using these optimization parameters were applied to constrain the DeLamar project resources. The in-pit resources were further constrained by the application of a gold-equivalent cutoff of 0.17 g/t to all in-situ model blocks lying within the optimized pits that are coded as oxide or mixed, a 0.1

g/t gold-equivalent cutoff to all stockpile material, a 0.3 g/t gold-equivalent cutoff for in-situ blocks coded as non-oxide at DeLamar, and a 0.2 g/t cutoff for in-situ blocks coded as non-oxide at Florida Mountain. Gold-equivalent grades were used solely for the purpose of applying the resource cutoffs, are a function of metal prices (Table 1-1) and metal recoveries, with the recoveries varying by deposit and oxidation state (Table 1-2).

The total DeLamar project resources are summarized in Table 1-3.

Table 1-3 Total DeLamar Project Gold and Silver Resources

Type	Class	Tonnes	Au g/t	Au oz	Ag g/t	Ag oz
Oxide	Measured	6,313,000	0.36	74,000	16.9	3,427,000
	Indicated	42,346,000	0.35	471,000	13.4	18,291,000
	Inferred	11,132,000	0.28	99,000	7.8	2,795,000
	Meas + Ind	48,659,000	0.35	545,000	13.9	21,718,000
Mixed	Measured	10,043,000	0.42	136,000	21.8	7,032,000
	Indicated	60,136,000	0.35	672,000	15.0	29,010,000
	Inferred	8,533,000	0.27	74,000	8.4	2,302,000
	Meas + Ind	70,179,000	0.37	808,000	16.5	36,042,000
NonOxide	Measured	21,056,000	0.51	345,000	32.8	22,198,000
	Indicated	65,486,000	0.45	943,000	22.2	46,640,000
	Inferred	18,561,000	0.38	229,000	14.0	8,371,000
	Meas + Ind	86,542,000	0.46	1,288,000	24.7	68,838,000
Stockpiles	Measured	-	-	-	-	-
	Indicated	42,455,000	0.22	296,000	11.8	16,149,000
	Inferred	4,877,000	0.17	26,000	9.8	1,535,000
	Meas + Ind	42,455,000	0.22	296,000	11.8	16,149,000
Total Resources	Measured	37,412,000	0.46	554,000	27.2	32,657,000
	Indicated	210,424,000	0.35	2,381,000	16.3	110,091,000
	Inferred	43,101,000	0.31	428,000	10.8	15,002,000
	Meas + Ind	247,836,000	0.37	2,935,000	18.1	142,748,000

Notes:

1. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*
2. *Michael M. Gustin, C.P.G. and Principal Consultant for RESPEC, is a Qualified Person as defined in NI 43-101, and is responsible for reporting mineral resources in this technical report. Mr. Gustin is independent of Integra.*
3. *In-Situ Oxide and Mixed and all Stockpile mineral resources are reported at a 0.17 and 0.1 g AuEq/t cut-off, respectively, in consideration of potential open-pit mining and heap-leach processing.*
4. *Non-Oxide mineral resources are reported at a 0.3 g AuEq/t cut-off at DeLamar and 0.2 g AuEq/t at Florida Mountain in consideration of potential open pit mining and grinding, flotation, ultra-fine regrind of concentrates, and either Albion or agitated cyanide-leaching of the reground concentrates.*
5. *The mineral resources are constrained by pit optimizations.*
6. *Gold equivalent grades were calculated using the metal prices and recoveries presented in Table 14.18 and Table 14.19.*
7. *Rounding as required by reporting guidelines may result in apparent discrepancies between tonnes, grades, and contained metal content.*
8. *The effective date of the mineral resources is August 25, 2023.*
9. *The estimate of mineral resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*

The project mineral resources are inclusive of the mineral reserves discussed herein. The mineral reserve statement included herein has an effective date of January 24, 2022 and is unaffected by the mineral resource update included herein. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

1.6 Mineral Reserves

This technical report also includes the results of a PFS and mineral reserve statement on the DeLamar project included in the NI 43-101 technical report titled "Technical Report and Preliminary Feasibility Study for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated March 22, 2022 with an effective date of January 24, 2022.

Mr. Dyer, P.E., the responsible qualified person for the mineral reserve estimate in the aforementioned technical report and included in this technical report, reviewed the updated mineral resource model and determined that the updated mineral resource model does not materially change the mineral reserve statement included in the aforementioned technical report. Accordingly, the results of the PFS and the mineral reserve statement have been reproduced in this technical report and remain unaffected by the updated mineral resource. The PFS and mineral reserve statement have an effective date of January 24, 2022.

Mr. Dyer has used Measured and Indicated mineral resources as the basis to define mineral reserves for both the DeLamar and Florida Mountain deposits. Mineral reserve definition was done by first identifying ultimate pit limits using economic parameters and pit optimization techniques. The resulting optimized pit shells were then used for guidance in pit design to allow access for equipment and personnel. Mr. Dyer then considered mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, and governmental factors for defining the estimated mineral reserves.

The economic parameters and cutoff grades used in the estimation of the mineral reserves are shown in Table 1-4 The overall leaching process rate is planned to be 35,000 tonnes (38,581 tons) per day or 12,600,000 tonnes (13,889,123 tons) per year for both Florida Mountain and DeLamar oxide and mixed material. DeLamar leach processing will also include agglomeration. Initially only the oxide and mixed material will be processed, then starting in year 3, non-oxide will be processed through a plant constructed to operate at a rate of 6,000 tonnes (6,614 tons) per day or 2,160,000 tonnes (2,380,992 tons) per year.

The cutoff grades applied reflect the cost to process material along with G&A and incremental haulage costs. Note that royalties are built into the block values and are considered in determining whether to process the material. While the DeLamar non-oxide breakeven cutoff grade would be $11.44/t according to the applicable costs, a cutoff of $15.00 was assigned to enhance the project's economic performance.

Table 1-4 DeLamar and Florida Mountain Economic Parameters

	DeLamar			Florida Mnt			
	Oxide	Mixed	Non-Oxide	Oxide	Mixed	Non-Oxide	Units
Mining Cost	$ 2.00	$ 2.00	$ 2.00	$ 2.00	$ 2.00	$ 2.00	$/t Mined
Incremental Ore Haulage	$ 0.20	$ 0.20	$ 0.20	$ 0.20	$ 0.20	$ 0.20	$/t Processed
Process Cost	$ 3.00	$ 4.00	$ 11.02	$ 2.75	$ 3.50	$ 9.00	$/t Processed
G&A	$ 0.44	$ 0.44	$ 0.22	$ 0.45	$ 0.45	$ 0.25	$/t Processed
GMV Breakeven COG	$ 3.64	$ 4.64	$ 11.44	$ 3.40	$ 4.15	$ 9.45	$/t Processed
GMV COG Used	$ 3.65	$ 4.65	$ 15.00	$ 3.55	$ 4.20	$ 10.35	$/t Processed
Final Process Costs	$ 4.27	$ 4.29	$ 11.91	$ 2.98	$ 3.67	$ 10.60	$/t Processed

GMV = gross metal value; COG = cutoff grade.

Total Proven and Probable reserves for the DeLamar project from all pit phases are 123,483,000 tonnes at an average grade of 0.45 g Au/t and 23.27 g Ag/t, for 1,787,000 ounces of gold and 92,403,000 ounces of silver (Table 1-5). The mineral reserves point of reference is the point where material is fed into the crusher.

Table 1-5 Total Proven and Probable Reserves, DeLamar and Florida Mountain

	Classification	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	Block Value
Oxide	Proven	3,295	0.39	41	17.39	1,842	19.34
	Probable	31,486	0.37	375	15.24	15,426	17.93
	P&P	34,782	0.37	416	15.44	17,268	$ 18.06
Mixed	Proven	7,741	0.49	122	25.75	6,409	23.72
	Probable	49,718	0.40	637	17.29	27,632	18.29
	P&P	57,459	0.41	759	18.43	34,042	$ 19.02
Non-oxide	Proven	7,321	0.65	153	53.15	12,511	39.33
	Probable	23,921	0.60	459	37.16	28,582	33.81
	P&P	31,243	0.61	612	40.91	41,093	$ 35.11
Total	Proven	18,358	0.54	316	35.18	20,763	$ 29.16
	Probable	105,126	0.44	1,471	21.20	71,640	$ 21.71
	P&P	123,483	0.45	1,787	23.27	92,403	$ 22.82

Notes:

1. *All estimates of mineral reserves have been prepared in accordance with NI 43-101 and are included within the current Measured and Indicated mineral resources.*
2. *Thomas L. Dyer, P.E. for RESPEC, a division of RESPEC, in Reno, Nevada, is a Qualified Person as defined in NI 43-101, and is responsible for reporting Proven and Probable mineral reserves for the DeLamar Project. Mr. Dyer is independent of Integra.*
3. *Mineral reserves are based on prices of $1,650 per ounce Au and $21.00 per ounce Ag. The reserves were defined based on pit designs that were created to follow optimized pit shells created in Whittle. Pit designs followed pit slope recommendations provided by RESPEC.*
4. *Reserves are reported using block value cutoff grades representing the cost of processing:*
5. *Florida Mountain oxide leach cutoff grade value of $3.55/t.*
6. *Florida Mountain mixed leach cutoff grade value of $4.20/t.*
7. *Florida Mountain non-oxide mill cutoff grade value of $10.35/t.*
8. *DeLamar oxide leach cutoff grade value of $3.65/t*
9. *DeLamar mixed leach cutoff grade value of $4.65/t.*
10. *DeLamar non-oxide mill cutoff grade value of $15.00/t.*
11. *The mineral reserves point of reference is the point where is material is fed into the crusher.*
12. *The effective date of the mineral reserves estimate is January 24, 2022.*
13. *All ounces reported herein represent troy ounces, "g Au/t" represents grams per gold tonne and "g Ag/t" represents grams per silver tonne.*
14. *Columns may not sum due to rounding.*
15. *The estimate of mineral reserves may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*
16. *Energy prices of US$2.50 per gallon of diesel and $0.065 per kWh were used.*

1.7 Mining Methods

The PFS reproduced and presented in this report considers open-pit mining of the DeLamar and Florida Mountain gold-silver deposits. Mining will utilize 23-cubic meter (30-cubic yard) hydraulic shovels along with 13-cubic meter (16.7-cubic yard) loaders to load 136-tonne capacity haul trucks. The haul trucks will haul waste and ore out of the pit and to dumping locations. Due to the length of ore hauls, the ore will be stockpiled near the pits followed by loading into a Railveyor system which will convey the ore into a crusher. The Railveyor system will be supplemented with haul trucks on an as needed basis.

Waste material will be stored in waste-rock storage facilities ("WRSFs") located near each of the Florida Mountain and DeLamar deposits, as well as backfilled into pits where available. The exception is the Milestone pit, from which waste material will be fully utilized for construction material for the tailing storage facility ("TSF").

Production scheduling was completed using Geovia's MineSched™ (version 2021) software. Proven and Probable reserves along with waste material inside pit designs previously discussed were used to schedule mine production. The production schedule considers the processing of DeLamar and Florida Mountain oxide and mixed material by crushing and heap leaching, with some of the DeLamar material requiring agglomeration prior to leaching. DeLamar and Florida Mountain non-oxide material would be processed using flotation followed by cyanide leaching of the flotation concentrate.

An autonomous Railveyor light-rail haulage system will be used to transport ore from the open pits to the crusher facility. Utilizing the Railveyor system allows the opportunity to realize cost savings compared to typical truck haulage. This system, in conjunction with the planned solar and liquid natural gas electrical microgrid will reduce the overall fuel consumption and carbon footprint of the project.

The PFS has assumed owner mining instead of the more expensive contract mining. The production schedule was used along with additional efficiency factors, performance curves, and productivity rates to develop the first-principal 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 will be required in addition to the primary mining equipment.

1.8 Processing and Recovery Methods

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

- Lower-grade oxide and mixed materials will be processed by crushed-ore cyanide heap leaching; and
- Non-oxide material will be processed using grinding followed by flotation, and very fine grinding of flotation concentrate for agitated cyanide leaching.

Heap-leach and milling ores will be coming from both the Florida Mountain and DeLamar deposits. Pregnant solutions from the heap-leach operation and from the milling operation will be processed by the same Merrill-Crowe zinc cementation plant. Processing will start with heap leaching in the first two years of operation. Milling of higher-grade non-oxide ore will start in the third year of operation.

Both Florida Mountain and DeLamar oxide and mixed ore types have been shown to be amenable to heap-leach processing following crushing. Material will be crushed in three stages to a nominal size of 80% finer than (P_{80}) 12.7-millimeter (0.5 inches), at a rate of 35,000 tonnes per day. About 45% of DeLamar ore is expected to require agglomeration.

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. Pregnant 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 precipitate will be filtered, dried, and smelted to produce gold and silver doré bullion for shipment off site.

The milling process will start with primary crushing of the ore to a nominal P_{80} of 120 millimeter (4.72 inches), followed by grinding in a SAG mill-ball mill circuit to a P_{80} of 150 microns. The ball mill discharge will be pumped to hydrocyclones, with the hydrocyclone overflow advancing to flotation and the underflow returning to the ball mill.

The flotation circuit will produce a sulfide concentrate that will recover gold and silver from the ore. This flotation concentrate will be reground to a nominal P_{80} of 20 microns before being leached in agitated leach tanks. Pregnant solution will be separated using a CCD circuit that employs dewatering cyclones and thickeners. The pregnant solution is then sent to the Merrill-Crowe plant and gold smelting facility to produce gold and silver doré bullion.

The flotation tailing stream will be thickened and pumped to the tailing storage facility. The concentrate leach residue will be sent to cyanide destruction, then stored in a separate concentrate leach tailing storage facility.

1.9 Capital and Operating Costs

Table 1-6 summarizes the estimated capital costs for the project. The life of mine ("LOM") total capital cost is estimated as $589.5 million, including $307.6 million in preproduction capital (including working capital and reclamation bond) and $281.8 million for expansion and sustaining capital. Sustaining capital includes $30.8 million in reclamation costs. The estimated capital costs are inclusive of sales tax, engineering, procurement, and construction management ("EPCM") and contingency.

Table 1-7 shows the estimated LOM operating costs for the project. Operating costs are estimated to be $12.93 per tonne processed for the LOM. This includes mining costs, which are estimated to be $1.90 per tonne mined. The total cash cost is estimated to be $923 per ounce of gold equivalent and site level all-in sustaining costs are estimated to be $955 per ounce of gold equivalent.

Table 1-6 Capital Cost Summary

Mine	Pre-Production	Sustaining Yr 1 to Yr 17	Total LOM
Mining Equipment	$ 28,859	$ 88,544	$ 117,403
Pre-Stripping	$ 12,712	$ -	$ 12,712
Other Mine Capital	$ 1,919	$ 225	$ 2,144
Sub-Total Mine	$ 43,490	$ 88,769	$ 132,260
Processing			
Leach Pad Construction Cost	$ 42,296	$ 11,035	$ 53,331
Oxide Plant Construction	$ 165,198	$ 8,842	$ 174,040
Non Oxide Mill Construction	$ -	$ 132,005	$ 132,005
Tailings Storage Facility Construction	$ 3,836	$ 58,793	$ 62,629
Sub-Total Processing	$ 211,330	$ 210,675	$ 422,005
Infrastructure			
Power	$ 3,500	$ -	$ 3,500
Access Road	$ 8,957	$ -	$ 8,957
Other	$ 7,652	$ 974	$ 8,626
Sub-Total Infrastructure	$ 20,109	$ 974	$ 21,083
Owner's Costs	$ 7,001	$ -	$ 7,001
SUB-TOTAL	$ 281,930	$ 300,418	$ 582,349
Other			
Working Capital	$ 19,518	$ (19,518)	$ -
Cash Deposit for Reclamation Bonding	$ 6,167	$ (6,167)	$ -
Salvage Value	$ -	$ (23,729)	$ (23,729)
TOTAL	$ 307,615	$ 251,004	$ 558,620
Reclamation	$ -	$ 30,835	$ 30,835
Total Including Reclamation Costs	$ 307,615	$ 281,839	$ 589,454

Notes:

1. *Capital costs include contingency and EPCM costs.*
2. *Mining equipment includes cost of Railveyor.*
3. *Major mining equipment assumes financing by equipment vendor with 10% down; principal payments included under sustaining capital column and interest payments included in operating costs.*
4. *Sustaining capital shown in this table includes expansion capital (non-oxide plant) and principal payment of mining equipment leases (see note 3 above).*
5. *Working capital is returned in year 17.*
6. *Cash deposit = 20% of bonding requirement. Released once reclamation is completed.*
7. *Salvage value for mining equipment and plant.*

Table 1-7 Operating and Total Cost Summary

	US/Tonne	
LOM Operating Costs	**Mined**	**Processed**
Mining	$ 1.90	$ 6.09
Processing (HL + Mill)		$ 5.99
G&A		$ 0.86
Total Site Costs		**$ 12.93**
LOM Cash Costs and Site Level All-in Sustaining Costs	**By-Product** (1)	**Co-Product** (2)
Mining	$ 647	$ 418
Processing	$ 640	$ 414
G&A	$ 92	$ 59
Total Site Costs	**$ 1,379**	**$ 891**
Transport & Refining	$ 27	$ 17
Royalties	$ 23	$ 15
Total Cash Costs	**$ 1,429**	**$ 923**
Silver By-Product Credits	$ (931)	$ -
Total Cash Costs Net of Silver by-Product	**$ 498**	**$ 923**
Sustaining Capital	$ 50	$ 32
Site Level All-in Sustaining Costs	**$ 548**	**$ 955**

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.*

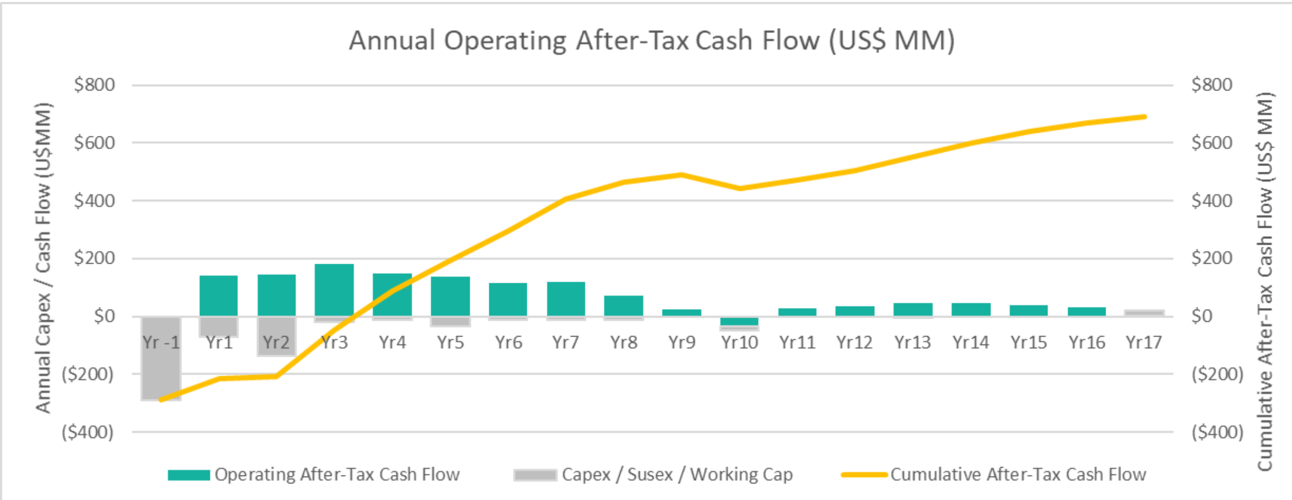
1.10 Economic Analysis

Economic highlights of the PFS for the DeLamar mining project include:

- Initial construction period is anticipated to be 18 months;
- After-tax net present value (“NPV”) (5%) of $407.8 million with a 27% after-tax internal rate of return (“IRR”) using $1,700 and $21.50 per ounce gold and silver prices, respectively;
- After-tax payback period of 3.34 years;
- Year 1 to 8 gold equivalent average production of 163,000 ounces (average 121,000 oz Au/year and 3,312,000 oz Ag/year);
- Year 1 to 16 gold equivalent average production of 110,000 ounces (average 71,000 oz Au/year and 3,085,000 oz Ag/year).
- After-tax LOM cumulative cash flow of $689.3 million; and
- Average annual after-tax free cash flow of $59.8 million during production.

Figure 1.1 shows the annual operating after-tax cash flow.

Figure 1-1 Annual Operating After-Tax Cash Flow

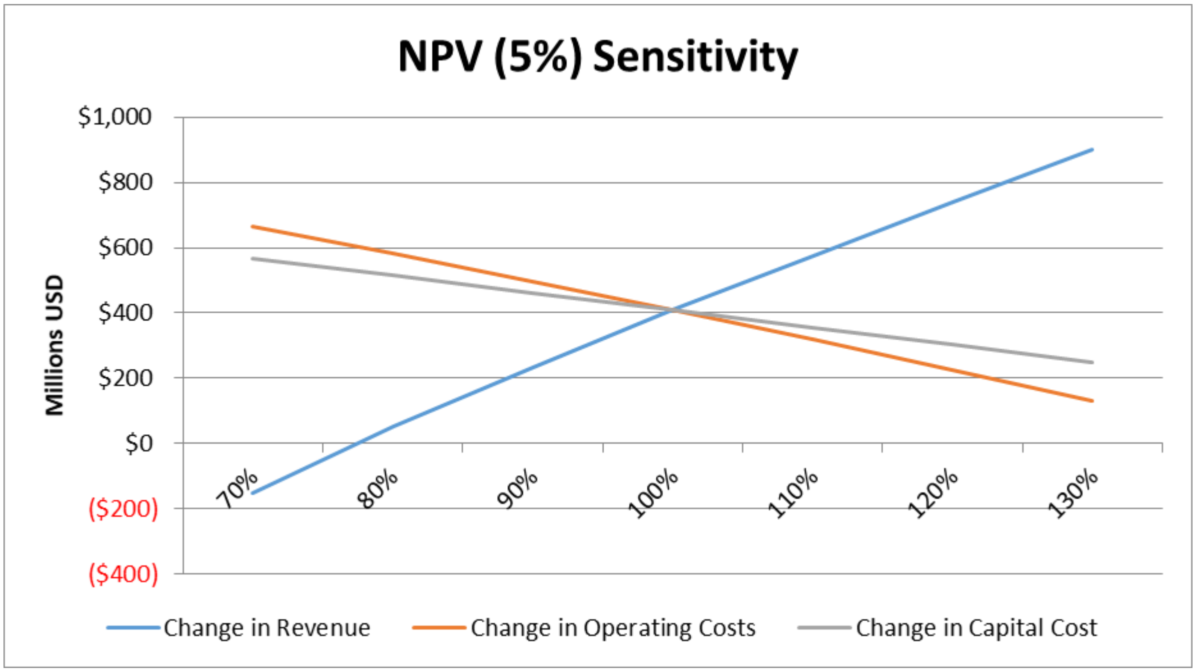


Economic sensitivities of the project to changes in metal prices were evaluated based on constant gold to silver ratios as shown in Table 1-8. The after-tax sensitivity to revenues, capital, and operating costs is shown in Figure 1.2.

Table 1-8 Project Sensitivity to Metal Prices

$/oz Au	$/oz Ag	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
$ 1,500	$ 18.97	$198,811	$123,406	$84,281	16%	4.30
$ 1,550	$ 19.60	$251,296	$167,213	$123,450	19%	3.94
$ 1,600	$ 20.24	$304,035	$211,159	$162,701	22%	3.72
$ 1,650	$ 20.87	$355,830	$254,247	$201,148	24%	3.52
$ 1,700	$ 21.50	$407,817	$297,519	$239,771	27%	3.34
$ 1,750	$ 22.13	$459,528	$340,561	$278,192	29%	3.19
$ 1,800	$ 22.76	$510,589	$383,015	$316,060	32%	3.05
$ 1,850	$ 23.40	$561,343	$425,183	$353,653	34%	2.93
$ 1,900	$ 24.03	$611,998	$467,275	$391,183	36%	2.83
$ 1,950	$ 24.66	$662,697	$509,428	$428,785	39%	2.73
$ 2,000	$ 25.29	$713,650	$551,851	$466,659	41%	2.64

Figure 1-2 After-Tax Sensitivity



1.11 Conclusions and Recommendations

Integra has advanced the DeLamar project to report herein the first Proven and Probable mineral reserves since mining by Kinross ceased in 1998. The reserves indicate a strong economic viability as shown with (i) after-tax NPV (5%) of $407.8 million with a 27% after-tax IRR using prices of $1,700 and $21.50 per ounce gold and silver, respectively; (ii) after-tax payback period of 3.34 years; and (iii) year 1 to 8 average production of 163,000 oz AuEq (121,000 oz Au and 3,312,000oz Ag), with year 1 to 16 average production of 110,000 oz AuEq (71,000 oz Au and 3,085,000 oz Ag). The total cash cost is estimated to be $923 per oz AuEq, with site-level all-in sustaining costs estimated to be $955 per oz AuEq.

The PFS total LOM gold production is estimated to be 1,154,000 ounces, with LOM average recovery of 72% for the heap leach and 51% for the mill. Silver production is estimated to be 50.0 million ounces, with an average LOM recovery of 37% for the heap leach and 75% for the mill.

With the updated mineral resource estimate included herein, project-wide Measured and Indicated resources, inclusive of Proven and Probable reserves, total 247,836,000 tonnes averaging 0.37 g Au/t (2,935,000 ounces of gold) and 18.1 g Ag/t (142,748,000 ounces of silver). Inferred resources total 43,101,000 tonnes at an average grade of 0.31 g Au/t (428,000 ounces of gold) and 10.8 g Ag/t (15,002,000 ounces of silver). Total Proven and Probable reserves for the DeLamar project from all pit phases are 123,483,000 tonnes at an average grade of 0.45 g Au/t and 23.27 g Ag/t, for 1,787,000 ounces of gold and 92,403,000 ounces of silver.

Metallurgical testing has shown that oxide and mixed mineralization types 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 with agglomeration required for a significant portion of the DeLamar area heap-leachable mineralization. Non-oxide mineralization from the Florida Mountain and DeLamar deposits is amenable to grinding followed by flotation, flotation concentrate regrind, and agitated cyanide leaching of the reground concentrate for recovery of gold and silver. The average silver recovery from the DeLamar non-oxide material is 75%, which leads to silver making a significant contribution to the project economics.

There is considerable exploration upside for both potential open-pit and underground mineable mineralization, the former at Florida Mountain and in Integra-held lands immediately outside of the bounds of the resources & reserves project area. The potential for underground mineable mineralization is particularly prospective in the Florida Mountain area. The potential of both areas and target types is supported by existing Integra drill results and warrants significant additional exploration investment.

1.11.1 Opportunities and Risks

There are many opportunities to improve the DeLamar project as discussed in Section 25, which in summary include (i) exploration both within the PFS project boundary and immediately outside of this boundary on other lands held by Integra that leads to additional open-pit and underground resources; (ii) additional metallurgical testing that results in improved process strategies and recoveries and/or decreased process cost; (iii) the completion of the drilling and metallurgical testing of stockpile materials that determines their suitability for being mined and processed in a similar manner as the current oxide and mixed reserves; (iv) improved understanding of the geotechnical aspects of the deposits that allows for steepening of the current slope parameters; and (v) the development of strategies for electrification of operations to decrease costs and CO_2 emissions.

An additional opportunity is the possibility of developing the DeLamar project deposits through a heap-leach only operation, which would lower the LOM capital costs and lower the operating costs while maintaining a robust production profile. In this scenario, the decision to construct and initiate mill processing could be exercised at any time, providing the flexibility to respond to changing market conditions and thereby reduce project risk.

Project risks include: (i) heap-leach gold and silver recoveries from DeLamar mixed materials and mill gold recoveries from non-oxide materials are variable and may not fully achieve projected recoveries; (ii) elevated-clay material at the DeLamar area may adversely affect projected heap leach and/or mill recoveries; (iii) further geotechnical studies for leach pads may result in less favorable geotechnical parameters that could add costs and larger footprints of heap-leach pads; and (iv) the hydrogeology is not well understood at present, which could lead to higher than anticipated water-management costs.

1.11.2 Recommended Work Program

Additional work is recommended to advance the DeLamar project, including the optimization of mine planning, metallurgical recoveries, and infrastructure, as well as permitting work, with the goal of filing a BLM Plan of Operations by the end of 2023 and completing a feasibility study in 2024-2025. Additional drilling to support these activities will also be needed. The total cost for the recommended program is approximately $70.6 million (Table 1-9).

Table 1-9 Summary of Integra Estimated Costs for Recommended Program

Item	Estimated Cost US$
Exploration Drilling (2,000 meters)	1,200,000
Stockpile Drilling (1,500 meters)	$600,000
Metallurgical Testwork	$400,000
Engineering, Design	$3,500,000
Permitting	$2,200,000
Total	***$7,600,000***

2.0 INTRODUCTION AND TERMS OF REFERENCE

RESPEC Company LLC ("RESPEC") has supervised the preparation of this technical report of 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 area subject to the mineral resource update and PFS included in this technical report.

Integra is listed on the TSX Venture Exchange (TSX.V: ITR) and the NYSE American Exchange (NYSE:ITRG). This report draws from previous technical reports by Gustin and Weiss (2017), Gustin et al., (2019a; 2019b), and Dyer, et al., (2022) and has been prepared in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("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 an updated technical summary of the DeLamar gold-silver project, including the first-time estimate of stockpile mineral resources and an update of the in-situ resources.

This technical report also includes the results of a PFS and mineral reserve statement on the DeLamar project included in the NI 43-101 technical report titled "Technical Report and Preliminary Feasibility Study for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated March 22, 2022 with an effective date of January 24, 2022. The results of the PFS and the mineral reserve statement included therein and reproduced in this technical report remain unaffected by the updated mineral resource included herein. The PFS and mineral reserve statement have an effective date of January 24, 2022.

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 reserves and 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. This work culminated in the estimation of mineral resources and reserves. References are cited in the text and listed in Section 20.0.

This report has been prepared under the supervision of Thomas L. Dyer, P.E. and Principal Engineer for RESPEC, Michael M. Gustin, C.P.G. and Principal Consultant for RESPEC, Jay R. Nopola, P.E. for RESPEC, Jack McPartland, Registered Member MMSA and Senior Metallurgist with McClelland Laboratories, Inc., Matthew Sletten, P.E. and Benjamin Bermudez, P.E. for M3 Engineering & Technology Corp. in Chandler, Arizona ("M3 Engineering", John Welsh, P.E., of Welsh Hagen and Associates in Reno, Nevada, John F. Gardner, P.E., of Warm Springs Consulting in Boise, Idaho, and Michael M. Botz, P.E. for Elbow Creek Engineering Inc., in Billings, Montana. Mr. Dyer, Mr. Gustin, Mr. Nopola, Mr. McPartland, Mr. Sletten, Mr. Bermudez, Mr. Welsh, Mr. Gardner, and Mr. Botz are Qualified Persons under NI 43-101 and have no affiliation with Integra except that of independent consultant/client relationships.

Table 2-1 lists the preparers of this report, all of which are qualified persons, as well as the sections of the report for which they are responsible and the date of their most recent site inspection, where applicable.

Table 2-1 Qualified Persons, Dates of Most Recent Site Visits, and Report Responsibilities

Company	Qualified Person	Professional Designation	Date of Most Recent Site Visit	QP Responsibilities by Report Section
RESPEC	Tom Dyer	P.E.	10/27/2020	1.7, 1.8, 1.10, 1.11, 1.12, 15) except 15.1), 16, 18.1, 18.5, 18.6, 18.9, 19, 21 (except 21.2, 21.6.1 through 21.6.5), 22, 25, 26
	Michael Gustin	C.P.G.	1/13/2023	1.1, 1.2, 1.3, 1.4, 1.6, 1.12, 2 through 12, 14, 20, 23 through 29
	Jay R. Nopola	P.E.	9/24/2020	15.1
McClelland Laboratories	Jack McPartland	QP MMSA	1/17/2019	1.5, 13
M3 Engineering	Matthew Sletten	P.E.	10/27/2020	18.7 and 18.8
	Benjamin Bermudez	P.E.	none	1.9, 17 (except 17.3.3, 17.3.4), 21.2 (except 21.2.6, 21.2.7), 21.6
Welsh Hagen Associates	John Welsh	P.E.	10/27/2020	17.3.3, 18.2, 18.3, 18.4, and portions of 21 and 25
Warm Springs Consulting	John F. Gardner	P.E.	none	18.5
Elbow Creek Engineering	Michael M. Botz	P.E.	none	17.3.2, 17.3.4

Mr. Gustin visited the project site on October 16, 17, and 18, 2018, October 15, 2020, October 27, 2020, October 19, 2022, and January 13, 2023, 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 during these visits. He also visited all relevant areas of the project, 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 visited the project site on October 27, 2020. Mr. Nopola visited the project site on September 24, 2020. 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. Benjamin Bermudez, P.E., for M3 Engineering, Mr. John Welsh, P.E., of Welsh Hagen Associates in Reno, Nevada, and Mr. Matthew Sletten of M3 Engineering contributed portions of Section 18, Infrastructure. Mr. Bermudez contributed to Section 21, Capital and Operating Costs. Mr. Sletten, Mr. Bermudez and Mr. Welsh are Qualified Persons under NI 43-101. Mr. Bermudez, Mr. Botz, and Mr. Gardner have not visited the project site. Mr. Welsh last visited the property on October 27, 2020. Mr. Sletten visited the property on October 27, 2020.

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. The authors have made such independent investigations as deemed necessary in their professional judgment to be able to reasonably present the conclusions discussed herein.

This report describes the estimated mineral resources and reserves for both the DeLamar and Florida Mountain areas. To avoid potential ambiguities, the term "DeLamar project" refers to the entire project, while "DeLamar", "DeLamar area", or "DeLamar deposit" and "Florida Mountain", or "Florida Mountain area", or "Florida Mountain deposit" refer to the individual areas.

The effective date of both the current mineral resources and this technical report is August 25, 2023. The PFS and mineral reserve statement have an effective date of January 24, 2022. Sections 15, 16, 17, 18, 19, 21, 22, 23 and 24 have been reproduced herein and have an effective date of January 24, 2022.

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	
1 cubic meter	= 264.172 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
AuEq	gold equivalent
cm	centimeters
core	diamond core-drilling method
ºC	degrees centigrade
CAD$	Canadian dollars
CO2e	carbon dioxide equivalent
dMTPH	dry metric tonne per hour
°F	degrees Fahrenheit
ft	foot or feet
gpm	gallons per minute
g/t	grams per tonne
ha	hectares
h or hr	hour
ICP	inductively coupled plasma analytical method
in.	inch or inches
kg	kilograms
km	kilometers
kPa	kilopascals

kt	kilotonnes
ktpd	metric kilotonnes per day
kV	kilovolt
kWh	kilowatt hour
l or L	liter
lbs	pounds
µm	micron
m	meters
m^3	cubic meters
Ma	million years old
mi	mile or miles
mm	millimeters
NPV	net present value
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 or mt	metric tonne or tonnes
TPD	metric tonnes per day
tph	metric tonnes per hour
ton	Imperial short ton
U.S.	United States of America
MW	megawatt
MWh	megawatt hours
XRD	x-ray diffraction
XRF	x-ray fluorescence

3.0 RELIANCE ON OTHER EXPERTS

The authors 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 lands 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 a limited Updated Title Report review (dated March 8, 2022), as well as 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 employees. Section 4.4, which pertains to environmental permits and liabilities, and Section 20, which discusses environmental permitting and related aspects of the project, were originally prepared by Integra's environmental and permitting team and subsequently reviewed and finalized by Mr. Gustin.

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, 4, and 20.

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, a mineral lands consultant for Integra. Mr. Devenyns prepared a Limited Title Report on the unpatented claims dated August 15, 2017 and Perkins Coie LLP prepared a Limited Due Diligence Review dated August 21, 2017. On March 2 and March 8, 2018, Perkins Coie LLP provided RESPEC information concerning the Banner and Empire claims at Florida Mountain. A limited Updated Title Report review of the property was prepared by Perkins Coie LLP dated February 14, 2023.

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

The authors 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 may be disclosed in this report.

4.1 Location

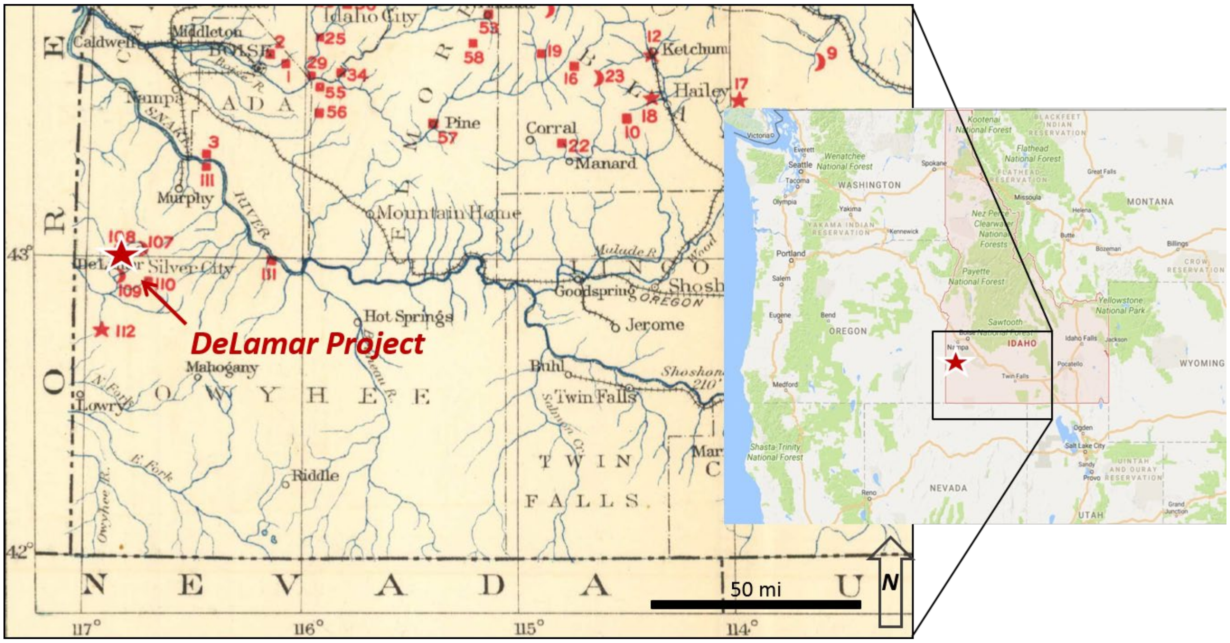
Integra's DeLamar gold-silver project is located in southwestern Idaho in Owyhee County, 80 kilometers (50 miles) southwest of the city of Boise, just west of the historical mining town of Silver City (Figure 4.1). The property is centered at approximately 43°00′48″N, 116°47′35″W, within the historical 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.

4.2 Land Area

The DeLamar project includes 790 unpatented lode, placer, and mill site 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 8,673 hectares (21,431 acres) 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 through 36 of Township 4 South, Range 4 West;
- Sections 25, 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.

Figure 4-1 Location Map, DeLamar Gold – Silver Project
(modified from Hill and Lindgren, 1912; red numbers refer to 1912 mining districts)

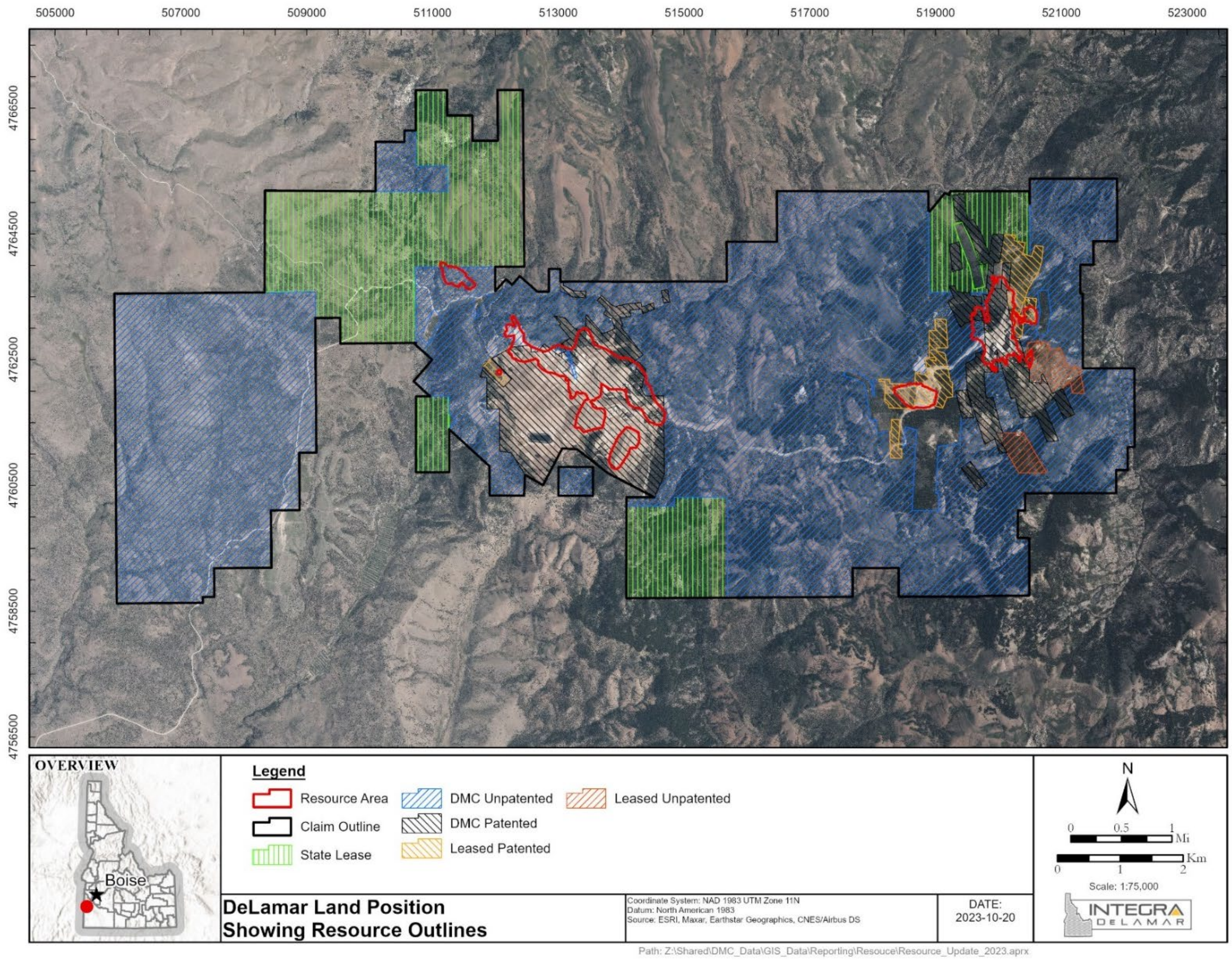


A listing of the patented and unpatented claims and leasehold interests that are included in the property is provided in Appendix A, Parts 1 through 6. Integra represents that the list of claims and leasehold interests in Appendix A is complete to the best of its knowledge as of the effective date of this report. Included in Appendix A, Part 1, are seven Idaho Department of Lands leases that have been 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, 2024. The current annual holding costs for the DeLamar project unpatented mining claims are estimated at $138,680 (Table 4-1), including the county recording fees.

Figure 4-2 Property Map for the DeLamar Project



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 $473,244.

The reviews by Mr. Devenyns and Perkins Coie LLP have not identified any known significant 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.

Table 4-1 Summary of Estimated Land Holding Costs for the DeLamar Project

Annual Fee Type	Amount
Unpatented Claims BLM Maintenance Fees	$ 138,600
Unpatented Claims County Filing Fees	$ 80
Estimated Holding Costs for Unpatented Mining Claims	**$ 138,680**
Access, Pipeline, Land Agreement Fees	$ 253,333
Owyhee County Patented Claims Taxes	$ 5,849
Patented Claims Agreement Fees	$ 48,100
State Lands Lease (annual rental and advanced minimum royalty payments)	$ 27,282
Total Estimated Annual Holding Taxes and Fees	**$ 473,244**

DMC holds the surface rights to the patented claims it owns and has leased, subject to various easements and other reservations and encumbrances. DMC has rights to use the surface of the unpatented mining claims for mining related purposes through September 1, 2024, 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. DMC holds surface rights to the areas it has under lease in accordance with the terms of each lease. These surface rights are considered sufficient for the exploration and mining activities proposed in this report, subject to regulation by the BLM and State of Idaho.

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 eight lease agreements that include 2% to 5.0% NSR obligations (referred to as Leases A through H in Figure 4.2, and Party A through G in Table 4-2) that apply to 33 of the patented claims and five unpatented claims. These claims are located within portions of Sections 1, 2, 4, 6, 11, and 12 of Township 5 South, Range 4 West; Sections 6 and 7 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 1,561 hectares (3,857 acres) leased from the State of Idaho under seven separate State Mineral Leases that are subject to a 5.0% net smelter returns production royalty (Table 4-2), plus annual lease fees of $27,282 (Table 4-1) The state lease E600067 has an expiration date of February 28, 2028. The six other state leases have an expiration date of February 28, 2029. The State of Idaho leases include very small portions of both the DeLamar and Florida Mountain resources.

Kinross had 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. The Kinross royalty has since been transferred to Triple Flag.

A total of approximately 20% of the current Florida Mountain resources and reserves 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 Royalties
(from Integra, 2021 and 2023)

Owner	Number of Claims or Lease	Royalty
Triple Flag	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 seven 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
Party G	7 patented claims	2% NSR
Party H	4 unpatented claims	2% NSR to a maximum of $80,000

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 of 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

The 1977 – 1998 DeLamar open-pit mining operations included the DeLamar and Florida Mountain mining areas. The DeLamar area 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 historical 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 discharging to the environment.

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

- Tailing pond de-watered and capped with clay and soil;
- Four 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, 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 to allow for positive drainage;
- 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 year-round treatment with storage of treated water and discharge during the spring and summer months.

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 $3,276,078 remains with the Idaho Department of Lands ("IDL") and a reclamation bond of $100,000 remains with the IDEQ. Additional reclamation bonds in the total amount of $714,400 have been placed with the BLM for exploration activities and groundwater well installation on public lands. There are also reclamation bonds with the IDL in the total amount of $155,900 for exploration activities on IDL leased lands.

As of the date of this report, Integra is conducting a 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 26.0 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.

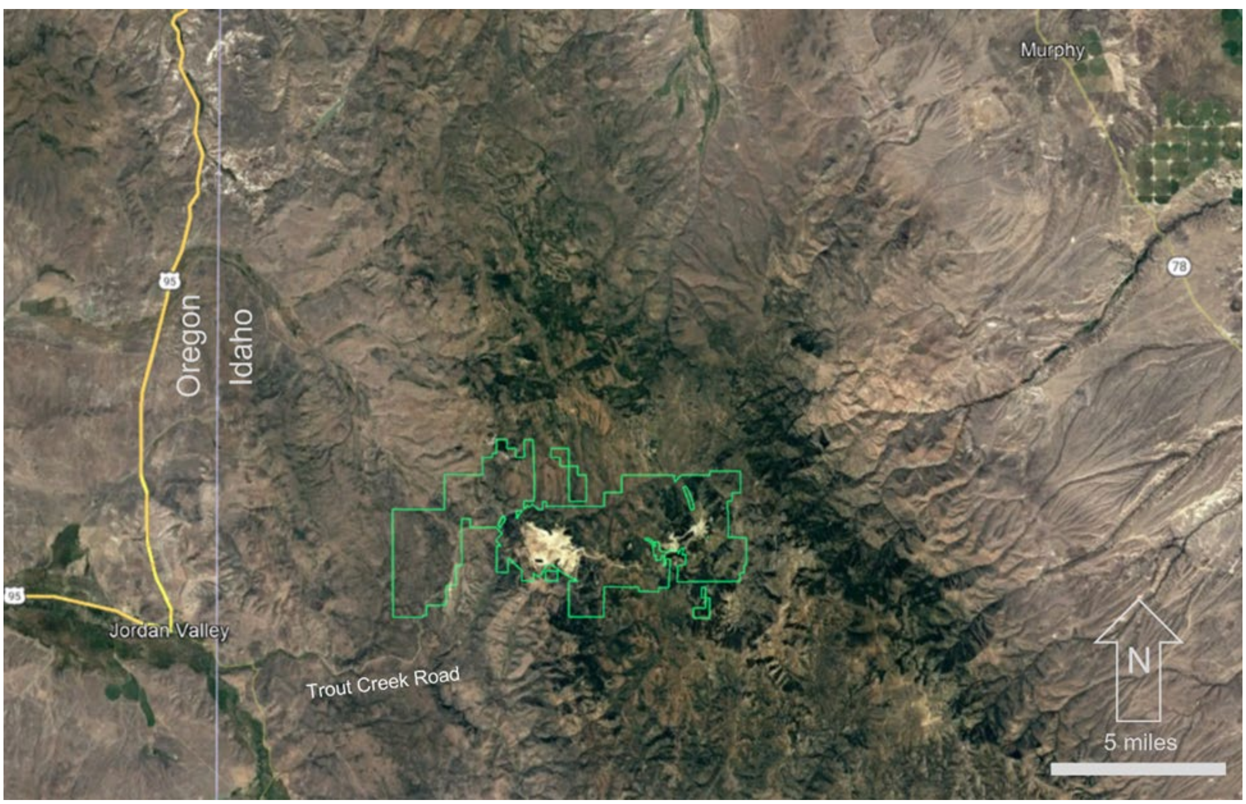
5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The information summarized in this section is derived from publicly available sources, as cited. Mr. Gustin has reviewed this information, and he believes this summary to be 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 (4.7 miles) to the Trout Creek Road (Figure 5.1). It is then another 39.4 kilometers (24.5 miles) travelling east on the gravel Trout Creek Road to reach the DeLamar mine tailing 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. Surface rights for access, exploration and mining are summarized in Section 4.2

Figure 5-1 Access Map for the DeLamar Project
(2022 property outline in green)



5.2 Physiography

The property is situated in rolling to mountainous terrain of the Owyhee Mountains at elevations ranging from about 1,525 meters (5,000 feet) to 2,350 meters (7,710 feet) 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. RESPEC is unaware of published historical temperature and precipitation data for the Owyhee Mountains. Summer maximum temperatures can reach 32°C (90°F) and winter minimum temperatures can be as low as -40°C (-40°F) according to Integra site personnel. Precipitation at the mine site is believed to average about 50 centimeters (20 inches) per year, most of which occurs as winter snowfall. Snow cover at the upper elevations can be one to two meters (3 to 6 feet) 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 (62 miles) 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 (18.6 miles) 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 (5.3 miles) 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 tailing storage areas. Developed water wells are present for mining and process requirements. Water required for the mining and process needs proposed in the PFS are discussed in Section 17.8 and Section 18.8. Areas for siting of the proposed waste-rock and tailings storage facilities, heap-leach pads, process plant, and energy needs are discussed in various portions of Section 18.0.

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 has reviewed this information and believes 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 portion of Integra's property that is the subject of this report. 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).

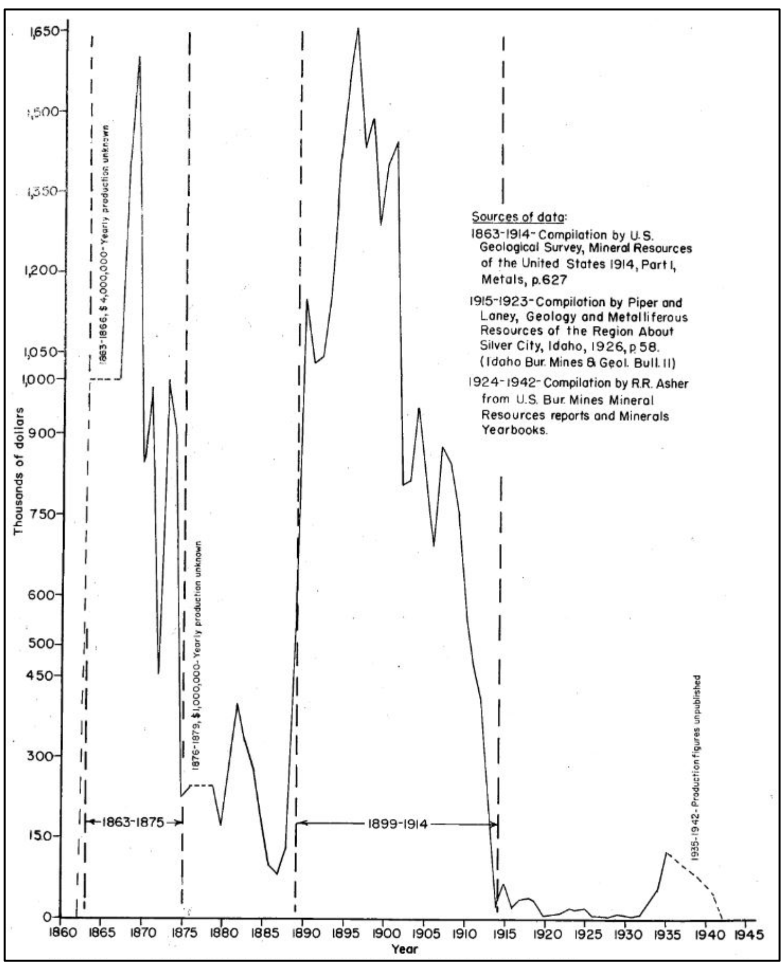
The general area of De Lamar, Florida Mountain, Silver City and War Eagle Mountain was known as the Carson mining district, which was larger than the current property controlled by Integra. There was only minor production from sporadic activity in the district at the War Eagle Mountain mines from 1876 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 RESPEC 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 RESPEC.

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.

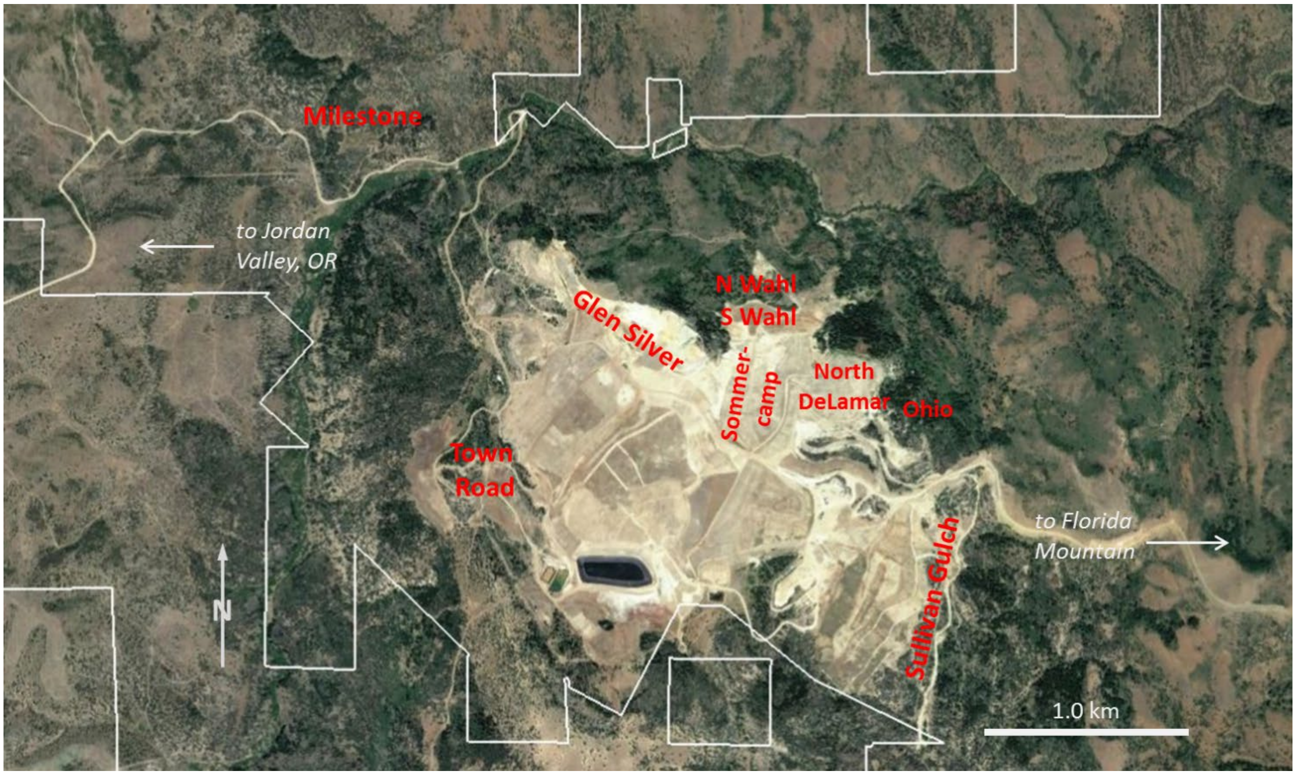
During 1969 and 1970, Earth Resources carried out trenching, sampling, and surface geological work, and drilled 39 conventional rotary drill holes at De Lamar Mountain. This resulted in the discovery of broad areas of near-surface silver-gold mineralization in the Sommercamp and Glen Silver zones, and 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.

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.

Figure 6-2 Aerial View, Zones of Exploration and Mining Since 1969 within the DeLamar Area
(produced by RESPEC, 2019)



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

From 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 on 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. Milling ceased in1999, 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 tailing 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 (5.2 mile) 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 RESPEC, 2019)

Figure 6-4 Photograph of the Reclaimed Florida Mountain (Stone Cabin) Mine Area

(view looking south-southeast)

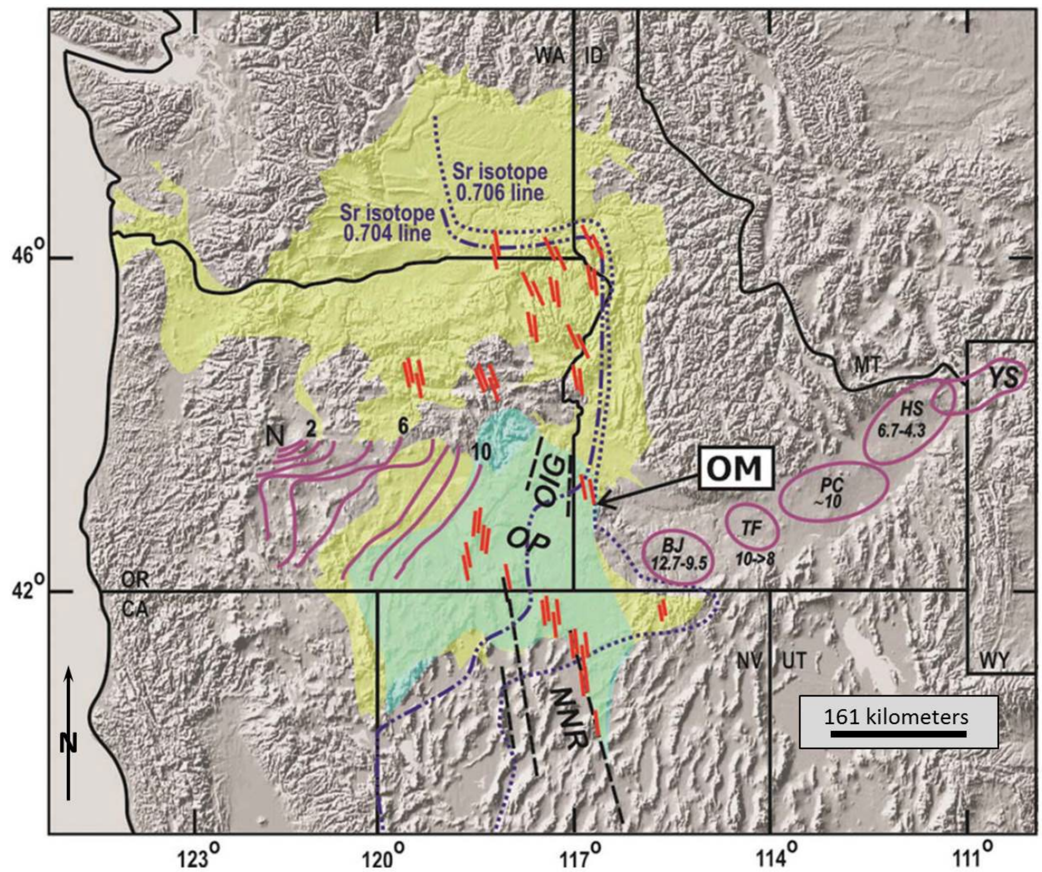
7.0 GEOLOGIC SETTING AND MINERALIZATION

The information presented in this section of the report is derived from multiple sources, as cited. Mr. Gustin has reviewed this information and believes 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).

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.

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).

7.2 Owyhee Mountains and District Geology

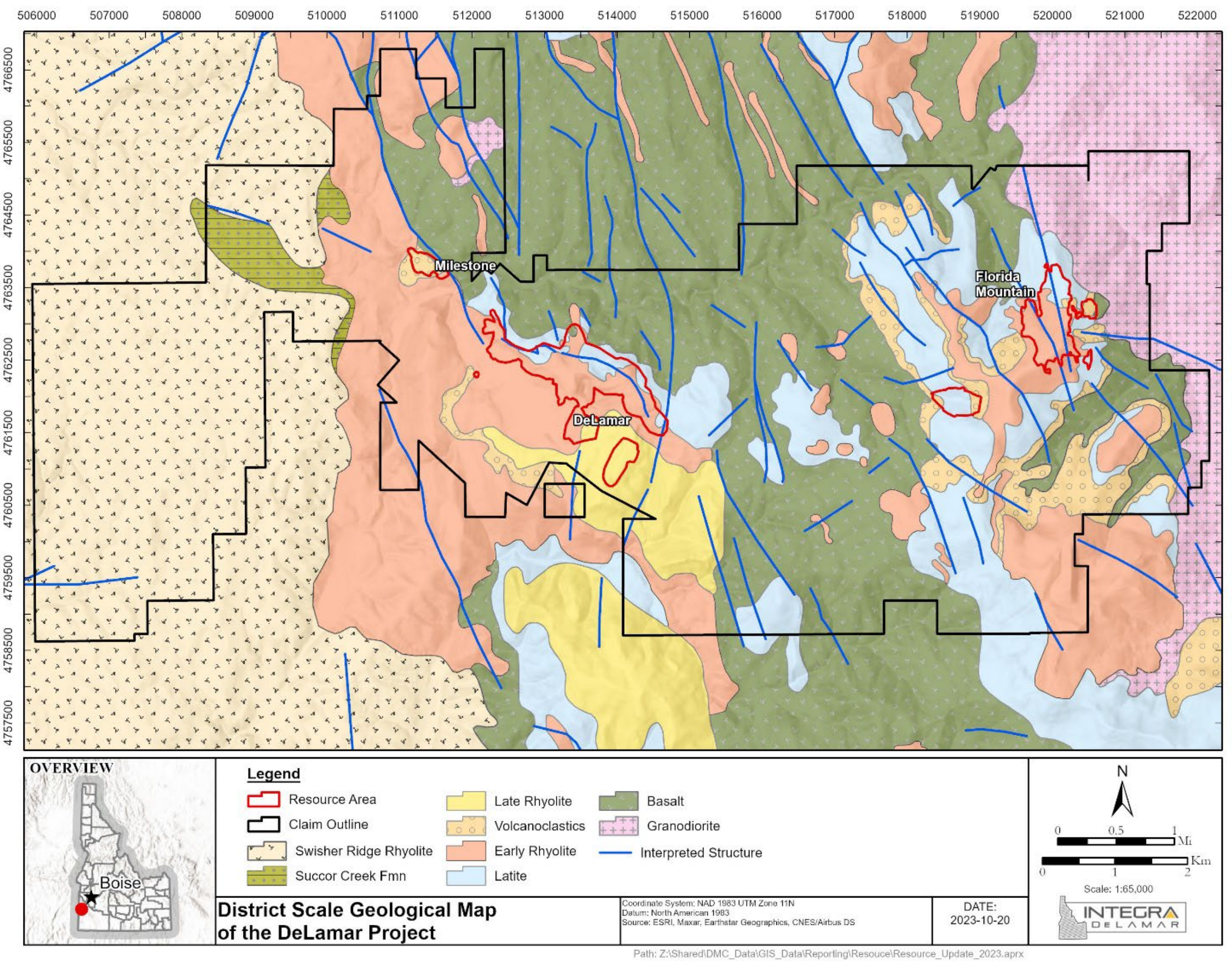
Five informal rock-stratigraphic sequences have been defined in the central Owyhee Mountains and the De Lamar – Silver City area (e.g., Ekren et al, 1981). From oldest to youngest these are the 1) Late Cretaceous Silver City granite; 2) mid-Miocene lower basalt; 3) mid-Miocene latite and quartz latite; 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). Integra's district geologic map is shown in (Figure 7.2).

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 (2,500 feet) 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 (1,800 feet) (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 457 meters (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 he 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).

Figure 7-2 Geologic Map of the Central Owyhee Mountains
(from Integra, 2023; black lines are property outline)



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.

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.

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

Figure 7-3 Land Position Map Showing Mineralized Zones

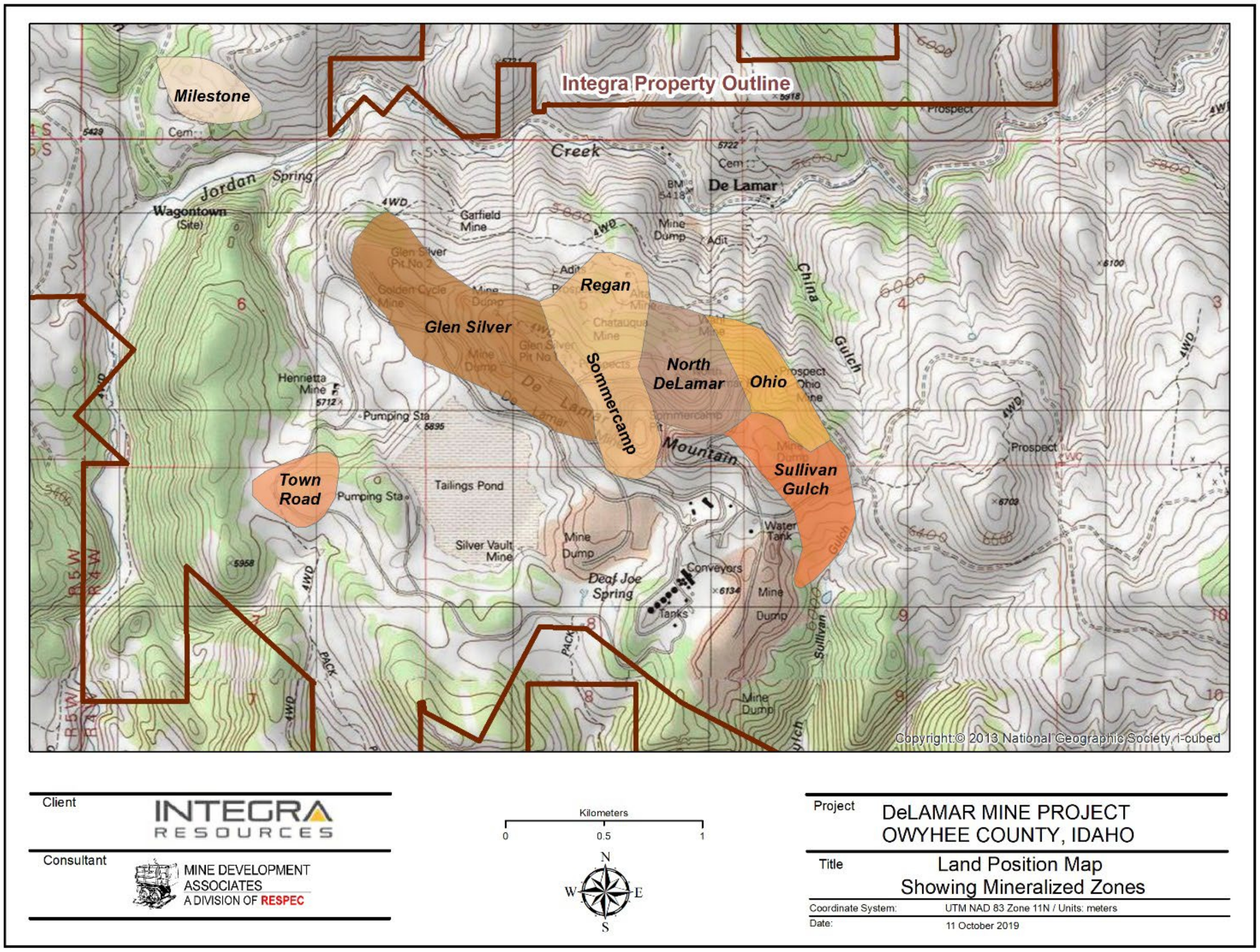
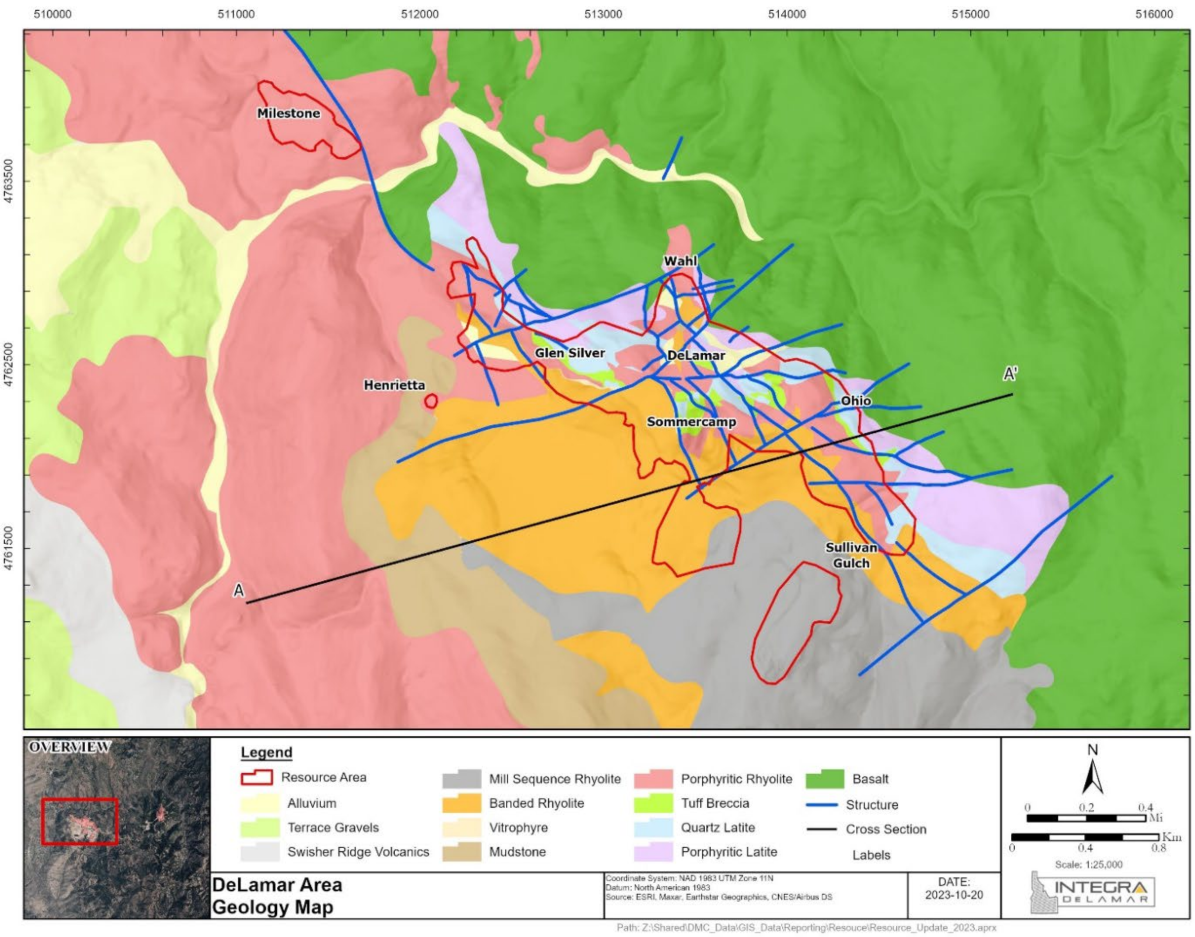
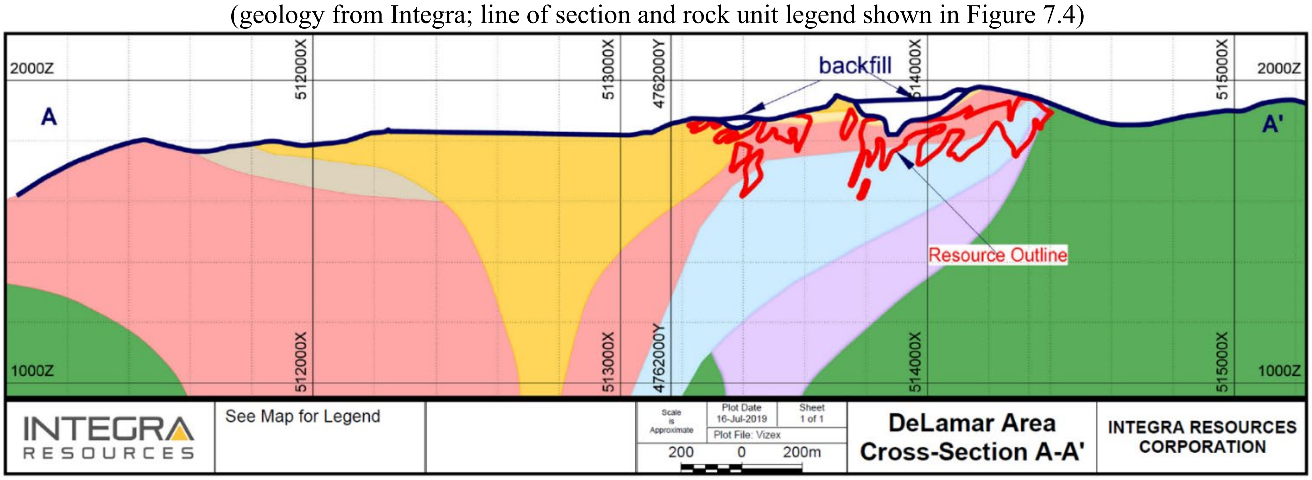


Figure 7-4 Integra Generalized 2018 DeLamar Area Geology
(geology from Integra, 2023)



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

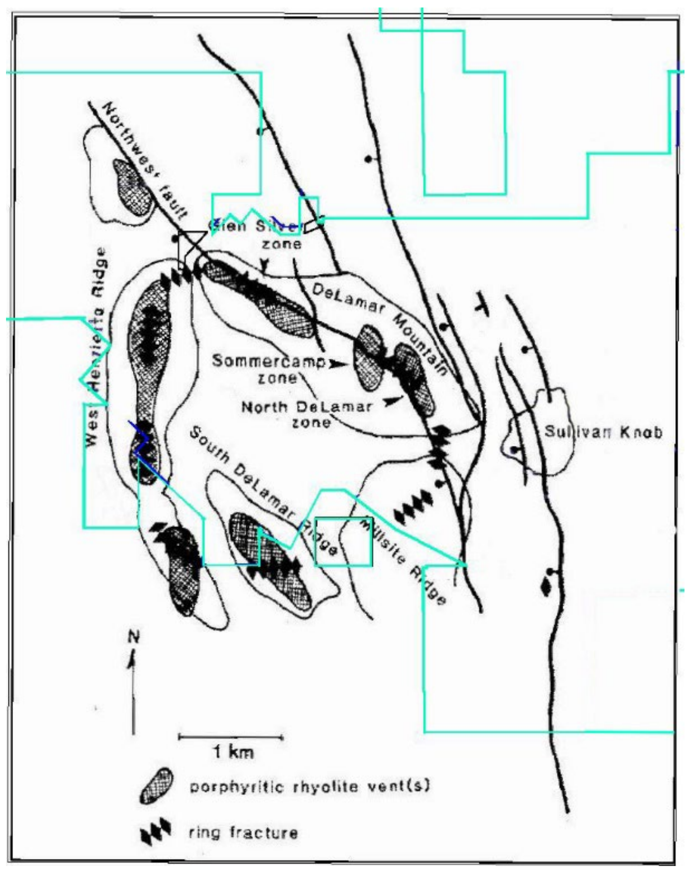
Figure 7-5 Integra 2018 Schematic Cross-Section, DeLamar Area
(geology from Integra; line of section and rock unit legend shown in Figure 7.4)



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 (46 feet) in thickness, strongly altered to clay, and includes fragmental volcanic layers of probable pyroclastic origin (Sillitoe, 2018; Hedenquist, 2018).

7.3.2 Florida Mountain 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. Integra's geologic map of the Florida Mountain area is shown in Figure 7.7.

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 granite (Figure 7.7) by the Integra 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. Integra's schematic cross section through Florida Mountain is shown in Figure 7.8.

The Earth Resources, NERCO and Integra geologists interpreted certain rocks at Florida Mountain to represent volcanic vents from which portions of the rhyolite flows and possibly tuffs were presumably erupted, 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.

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 has reviewed this information and believes it reasonably describes the mineralization as it is presently understood.

Figure 7-7 Geologic Map of Florida Mountain
(from Integra, 2023)

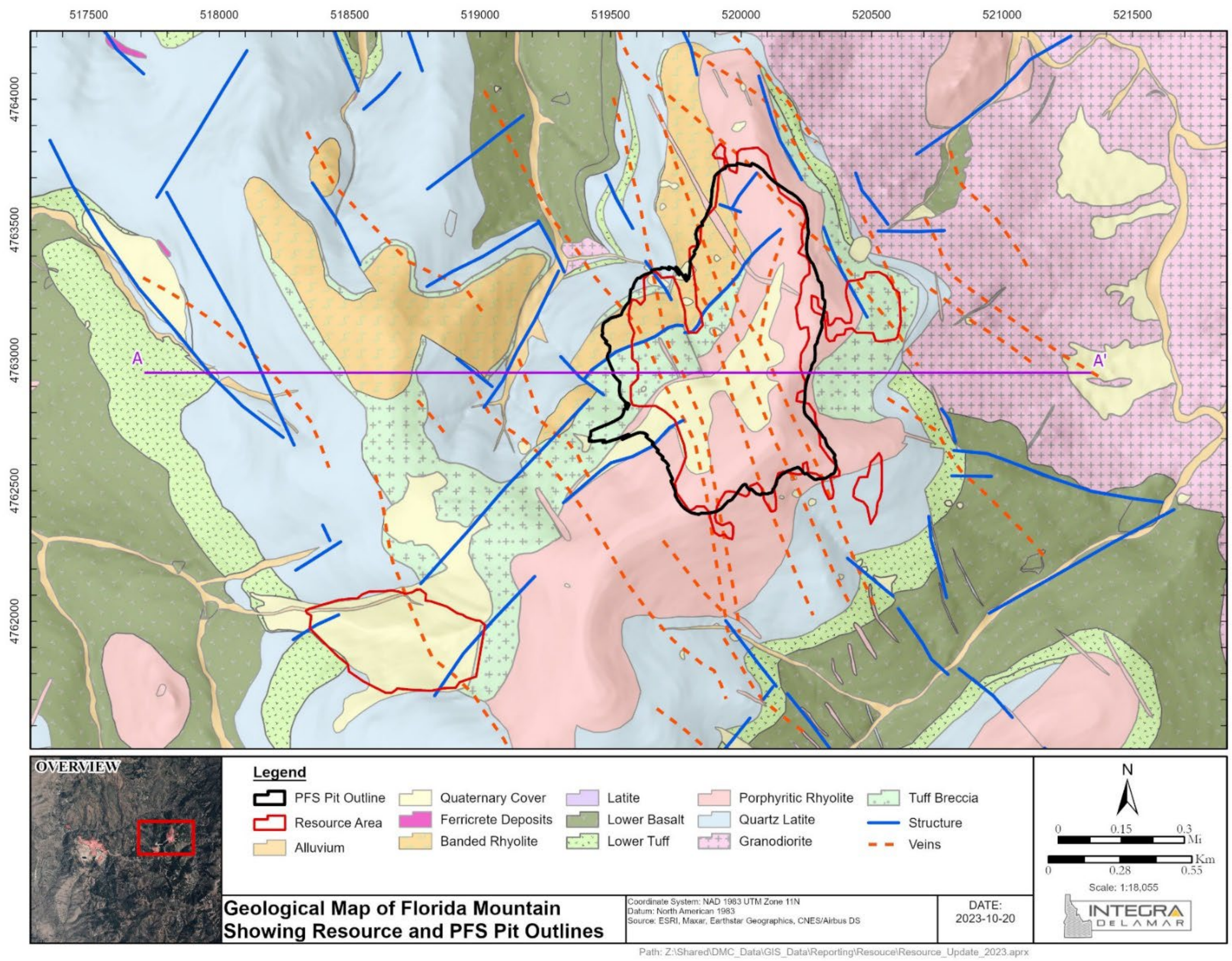
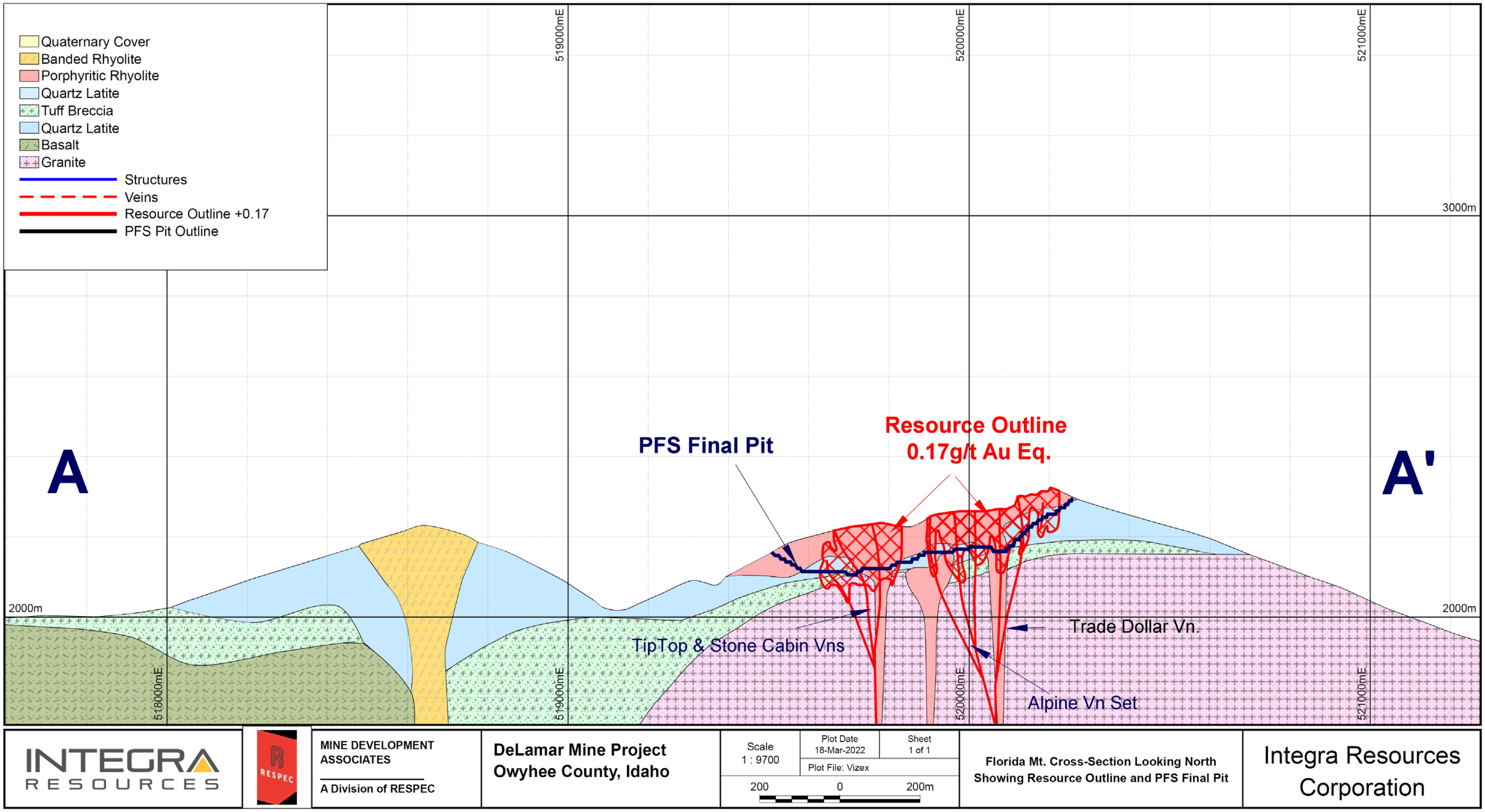


Figure 7-8 Schematic Florida Mountain Cross Section (Looking Northeast)
(from Integra, 2022)



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 several 10s of centimeters 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 about 23 meters (75 feet) in width, but more commonly were 6 meters (20 feet) 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 it 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 7.9 at the 4th level (elevation 1,902 meters) (6,240 feet).

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.

Figure 7-9 Veins of the Historical De Lamar Mine, Elevation 6,240 Feet
(from Asher, 1968; based on Piper and Laney, 1926)

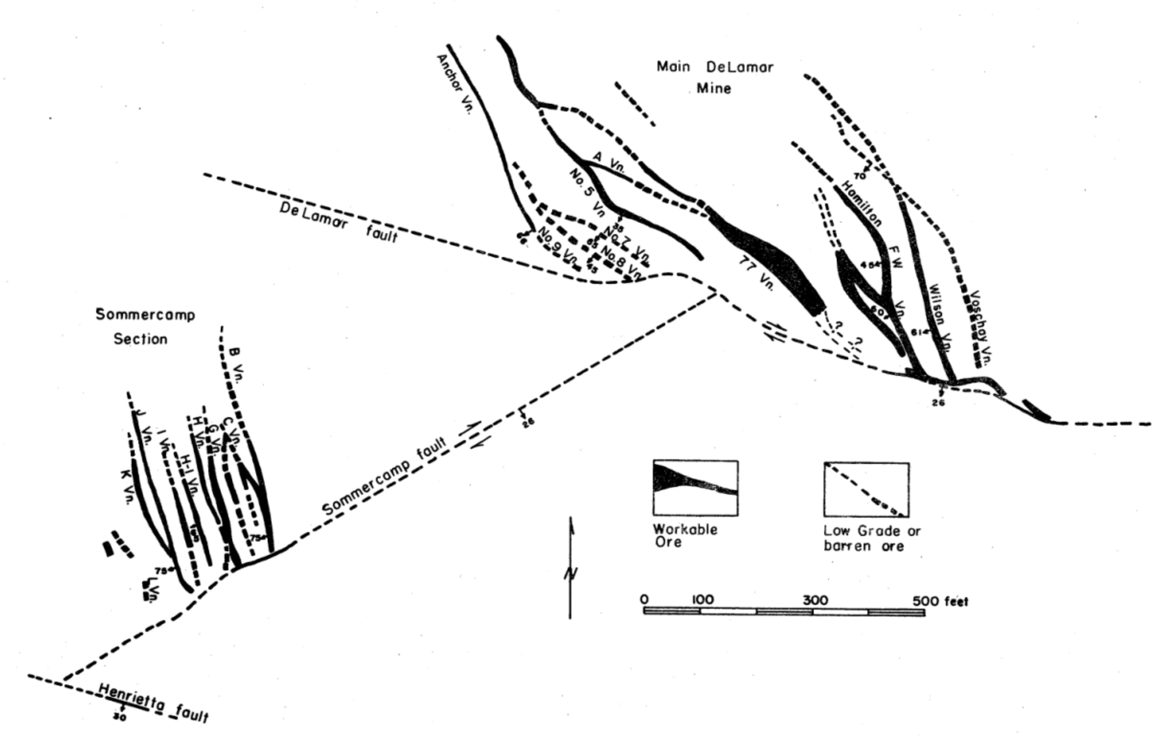


Figure 29 — Vein System of DeLamar Mine, 4th level, Silver City Region, Owyhee County, Idaho. (From Piper and Laney, 1926, PL X).

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

> *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 (1 to 24 feet). The overall width of the 77 vein was as much as about 23 meters (75 feet). During this period most of the production came from elevations above 1,786 meters (5,860 feet), but some stopes were as deep as the 12th level at 1,768 meters (5,800 feet). Although the 77 vein was found to persist to the 16th level at an elevation of 1,712 meters (5,617 feet), 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 has 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 to occur 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 (1 mile) 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 7.10.

The veins in Florida Mountain were mapped in greater detail in the 1970s and 1980s by Earth Resources, NERCO and later by Integra geologists (e.g., Figure 7.7), in part with the benefit of trenching and drilling. The most complete historical vein and geologic map that Mr. Gustin is 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.

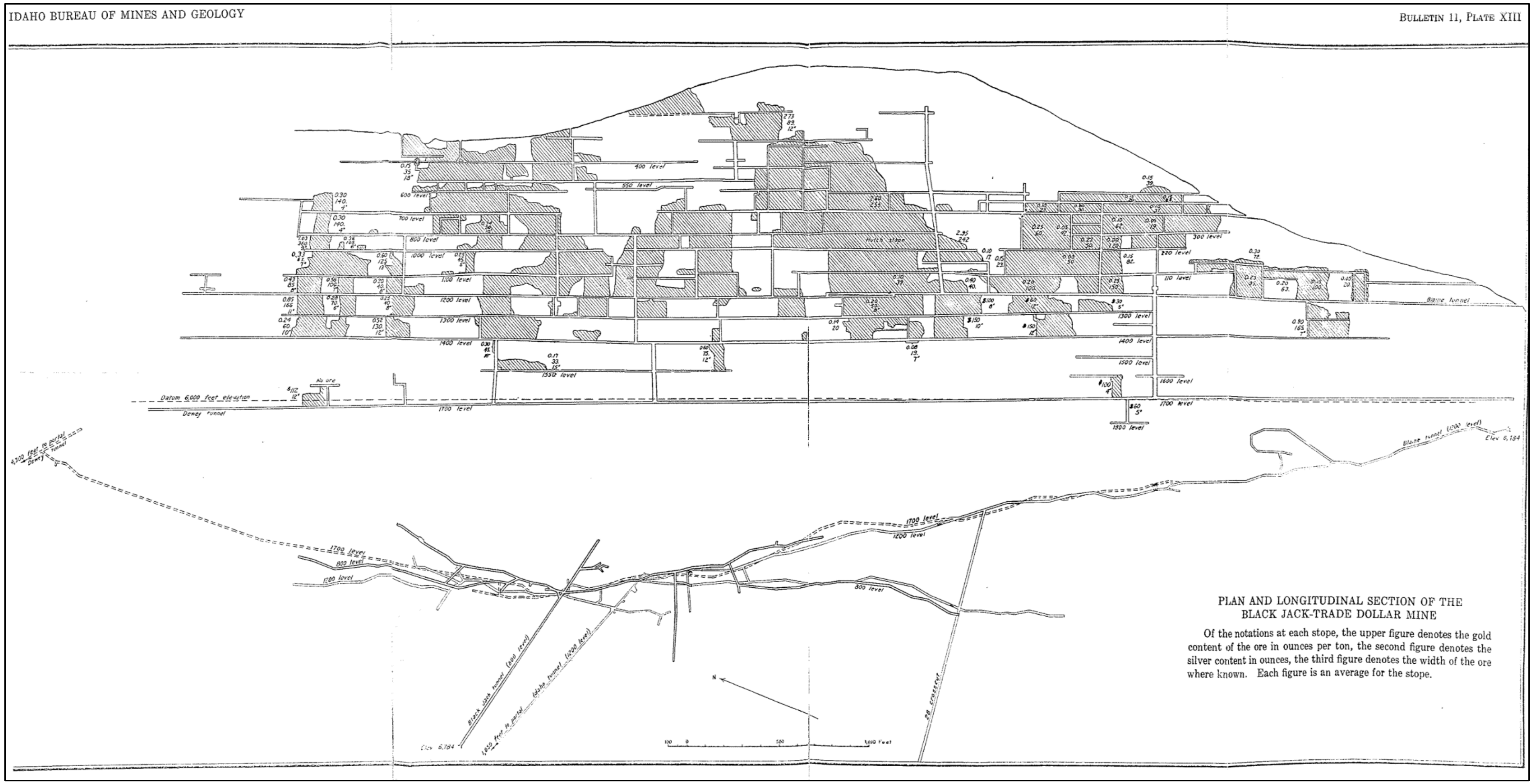
Mosser (1992) summarized the vein mineralization as follows:

> *"...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 (20 feet) 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). Information on the length, width, depth, and continuity of mineralization is summarized in various parts of Section 14.0.

Figure 7-10 Longitudinal Section of the Black Jack – Trade Dollar Mine
(from Piper and Laney, 1926)

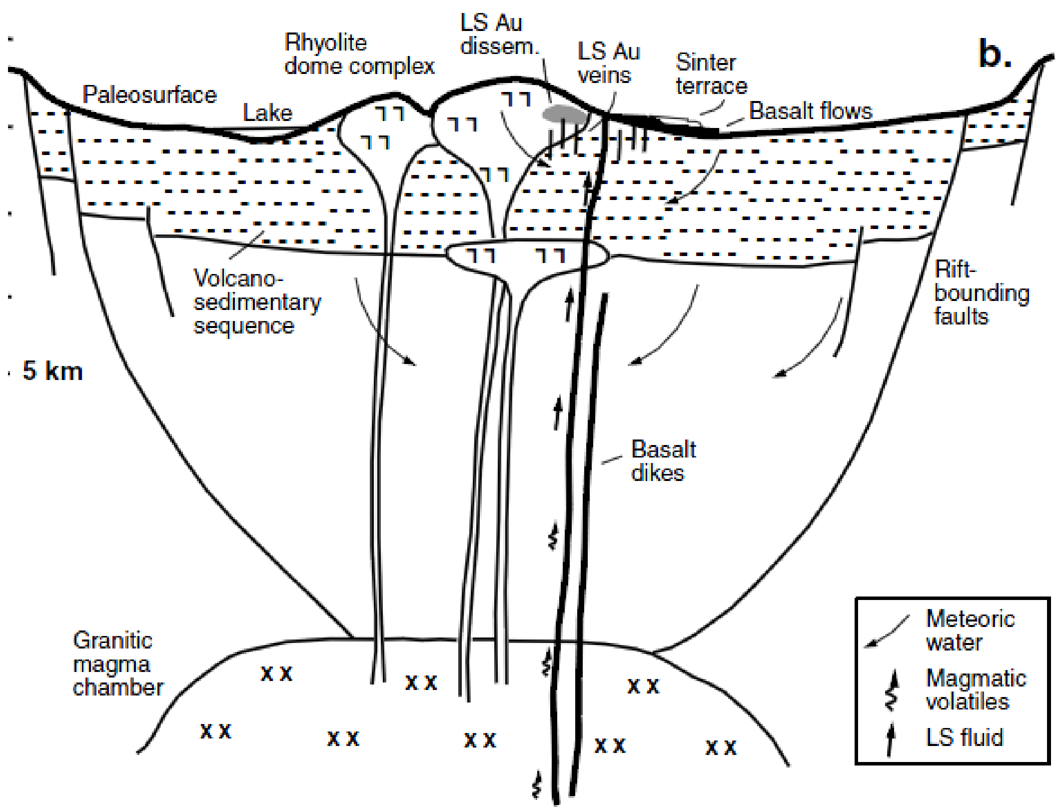


8.0 DEPOSIT TYPE

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 (455°F to 527°F). 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 (338°F to 464°F), 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.

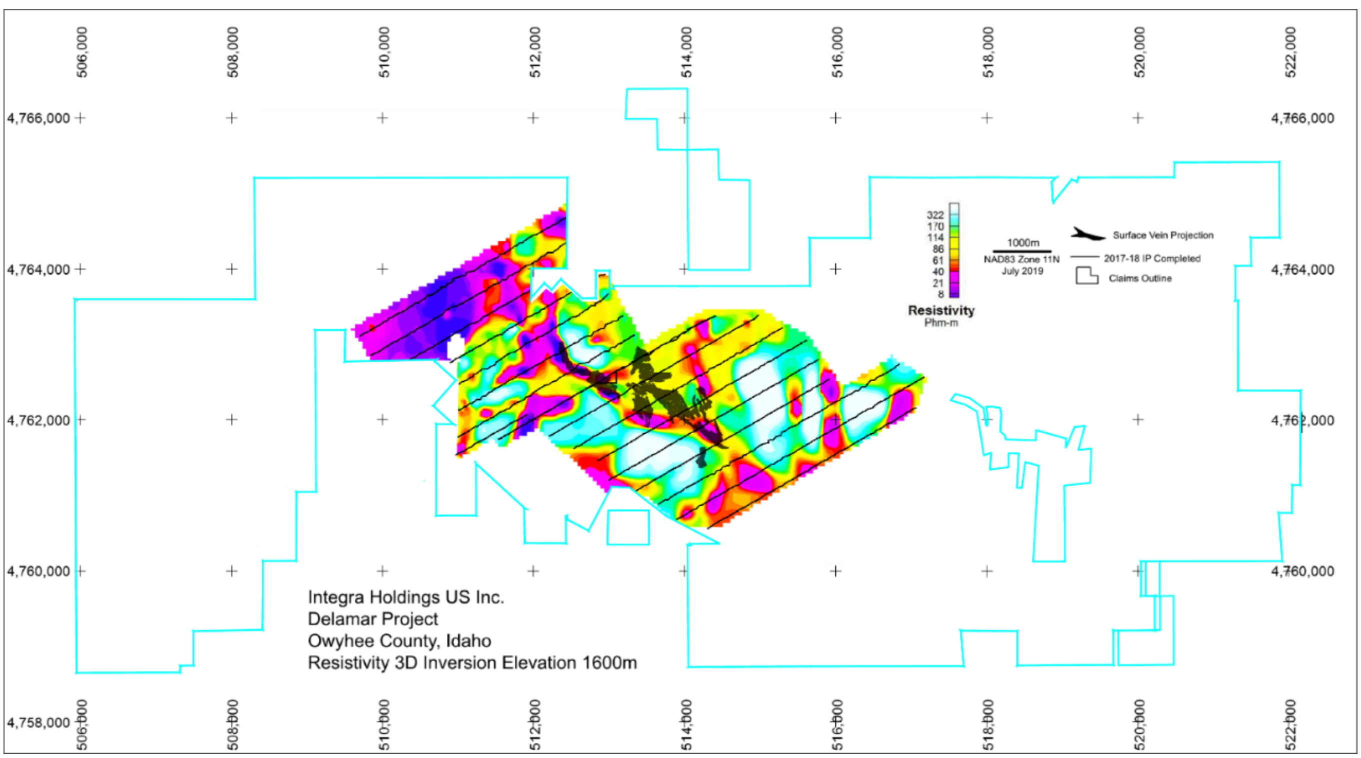
9.0 EXPLORATION

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

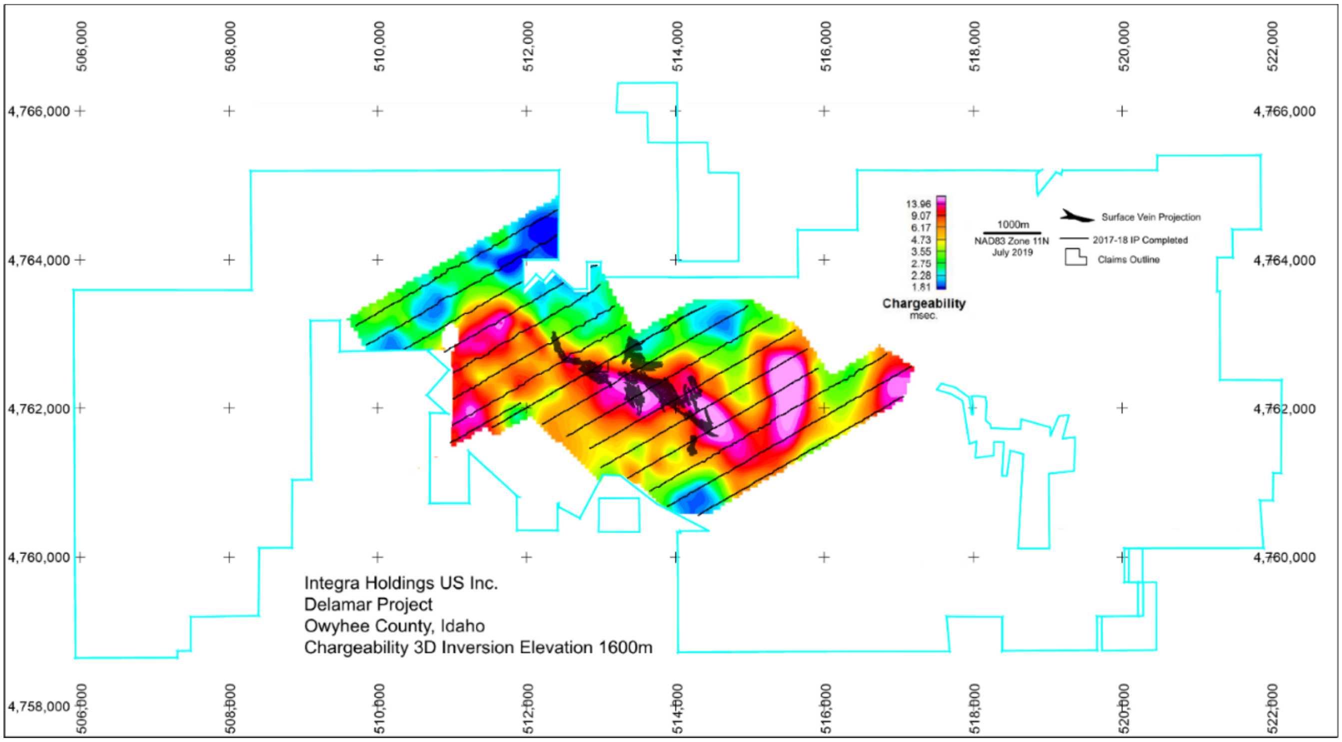
A Light Detection and Ranging ("LiDAR") topographic survey of the DeLamar and Florida Mountain areas was completed late in 2017. Integra also commissioned SJ Geophysics Ltd., of Delta, British Columbia, to conduct an Induced Polarization and Resistivity ("IP/RES") survey of six lines using the Volterra-2DIP distributed array system for a total of 22.4 line-kilometers (13.9 line-miles) 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 (25 line-miles). The IP/RES lines were spaced at 300 meters (984 feet) and utilized a potential dipole spacing with intermediate current spacing of 100 meters (328 feet). 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, claim outline of 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; claim outline of 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.

9.1.1 2019 Airborne Magnetic Survey

A helicopter high-resolution magnetic survey of the DeLamar – Florida Mountain area was conducted in 2019 by New Sense Geophysics Ltd., of Markham, Ontario. The survey used a 200-foot line separation at an average terrain clearance of 39 meters (128 feet). The term "high resolution" for this purpose implies a tight line spacing, stinger mounted magnetometer, sample rate of at least 50 Hz, low helicopter speed in dissected topography, and micro-leveling of the data. Basic processing of the 2019 data was done by the contractor and additional magnetic products including reduction to pole, various derivative products such as vertical derivative and analytic signal were prepared by Robert Ellis of Reno, Nevada, using Oasis Montaj software *(www.seequent.com)*. Both conventional susceptibility inversion (Li and Oldenburg, 1996) and magnetic vector inversion ("MVI") were used to generate a 3D voxel solid of the susceptibility distribution. The induced magnetic field direction at the time of the survey had an inclination of about 66.7° and a declination of about 13.6°. The direction of magnetization vector for high amplitude magnetic sources (i.e., Miocene basalts) defined in this inversion model varied from flat to +30° with declinations that between -100° and +100°. This confirms that remanent magnetization of the mafic rocks is present and the position and geometry of signatures with respect to the source locations in products like the reduction-to-pole and related products can be shifted. The amplitude component from the MVI inversion is the magnitude of the susceptibility accommodating remanence and induced magnetization and is referred to for convenience as susceptibility. This susceptibility model is used for the analysis of source locations of mafic and felsic intrusions, and intrusions within the granodiorite at Florida Mountain. The model was also used to better interpret structure.

9.1.2 2020 Induced Polarization and Resistivity Surveys

Induced polarization (chargeability) and resistivity data were collected at DeLamar from nine east-west lines spaced 300 meters (984 feet) apart and totaling 29 line-kilometers (18 line- miles) in 2020. Zonge International of Tucson, Arizona collected the data. A dipole-dipole configuration was utilized. 2D inversion models of the data, including the 2018 and 2018 distributed array data, were done using TS2Dip (*www.zonge.com/legacy/ModelIP.html*). A 3D inversion of the data produced a marginally deeper solid with the sacrifice of lost resolution at shallow depths. Consequently, the 2D model sections were gridded to 3D voxel solids and used to extract sections shown with geology interpreted from surface mapping and drilling for DeLamar and Florida Mountain.

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 (164-foot) intervals along lines spaced 300 meters (984 feet) apart, and 475 rock-chip samples were also collected.

During 2019 through 2023, Integra and contractor personnel collected 449 rock samples in the DeLamar, Milestone and Florida Mountain areas. Contractor personnel from Rangefront Geological ("Rangefront") of Elko, Nevada collected 298 soil samples in the DeLamar/Milestone area in 2019. A total of 2,332 soil samples were collected from the Florida Mountain area by Rangefront in 2019.

9.3 Geologic Mapping 2020 - 2021

Integra geologists carried out geologic mapping at a scale of 1:5,000 in 2020 and 2021. Approximately 6.25 square kilometers (2.4 square miles) were mapped in the DeLamar area. About 50 square kilometers (19.3 square miles) were mapped in the Florida Mountain area.

9.4 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 almost 2,500 historical RC drill holes and the re-logging data was added to the databases. This program included logging of oxidation types as oxide, mixed and non-oxide, the data for which had never 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. RESPEC 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 10^{th} 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.

9.5 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 (98.4-foot) 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. These cross sections and geologic information were summarized and interpreted as described in Section 14.2, 14.4, and 14.5. In essence, the significant results of Integra's exploration and interpretations of the exploration information are summarized in Section 14.2, 14.4, and 14.5 where these results and interpretations are applied to the estimation of the current mineral resources in this report.

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

RESPEC has records for a total of 372,888 meters (1,223,386 feet) drilled in 3,185 holes in the DeLamar and Florida Mountain portions of the property as summarized in Table 10-1. This includes Integra's drilling through the end of April 2023, i.e., the drill holes used in the current resource estimation. Drilling at the project has continued through 2023 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	2018-2023	436	57,809
Florida Mountain	2018-2023	124	39,289
Total Integra Drilling		560	97098
Grand Total	***1966 - 2023***	***3,185***	***372,888***

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, RESPEC 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 (904,823 feet). 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 (35,505 feet), or 3.9% of the overall meterage drilled. The median down-hole depth of all historical holes in the DeLamar area is 91 meters (298.6 feet), and the median depth in the Florida Mountain area is 123 meters (403.5

feet). 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 RESPEC 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 (4,521 feet) were drilled in five inclined core holes, but RESPEC 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 (7,555.9 feet, 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. RESPEC is unaware of the type(s) and size(s) of drill bits used.

Figure 10-1 Map of DeLamar Area Drill Holes

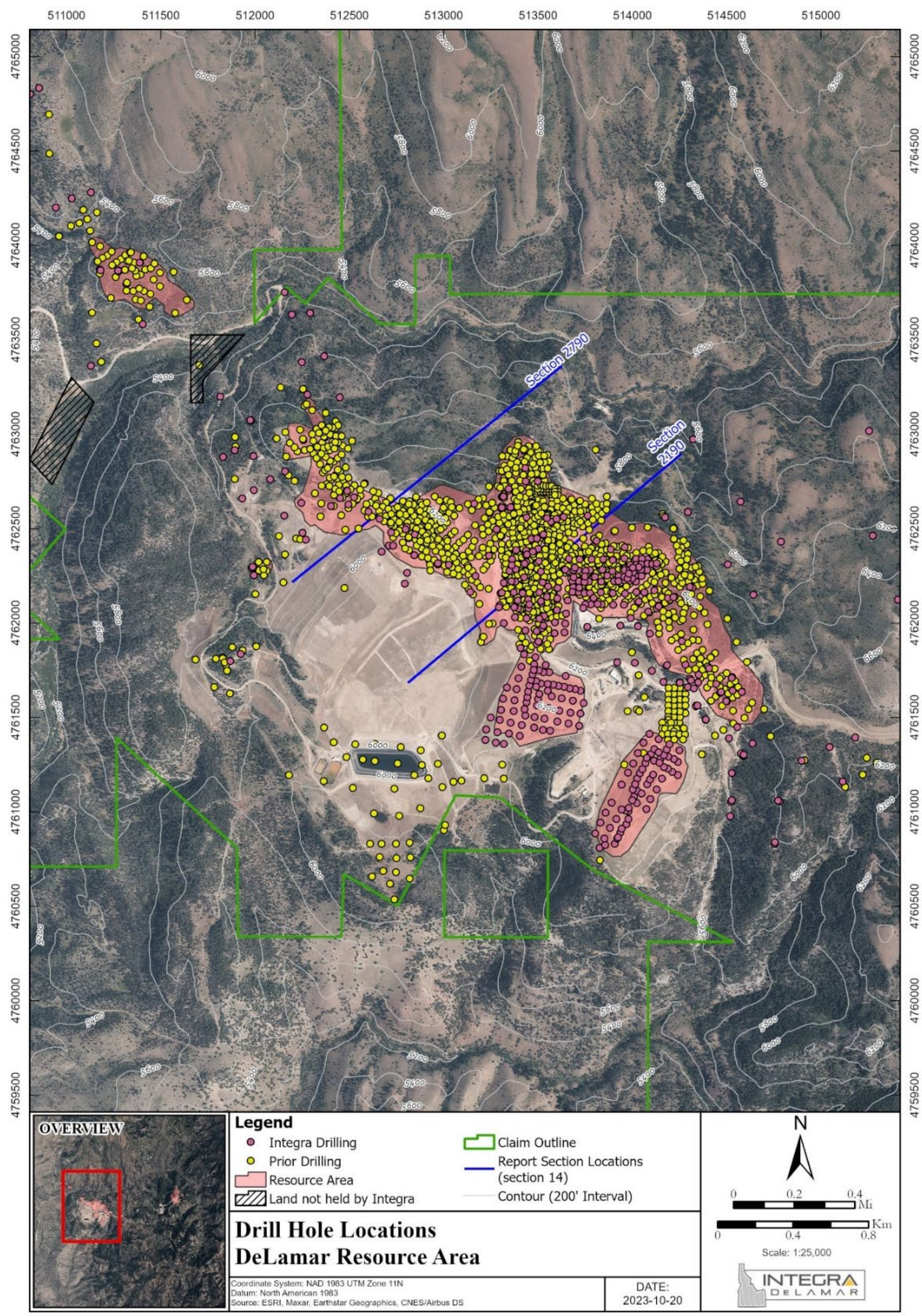
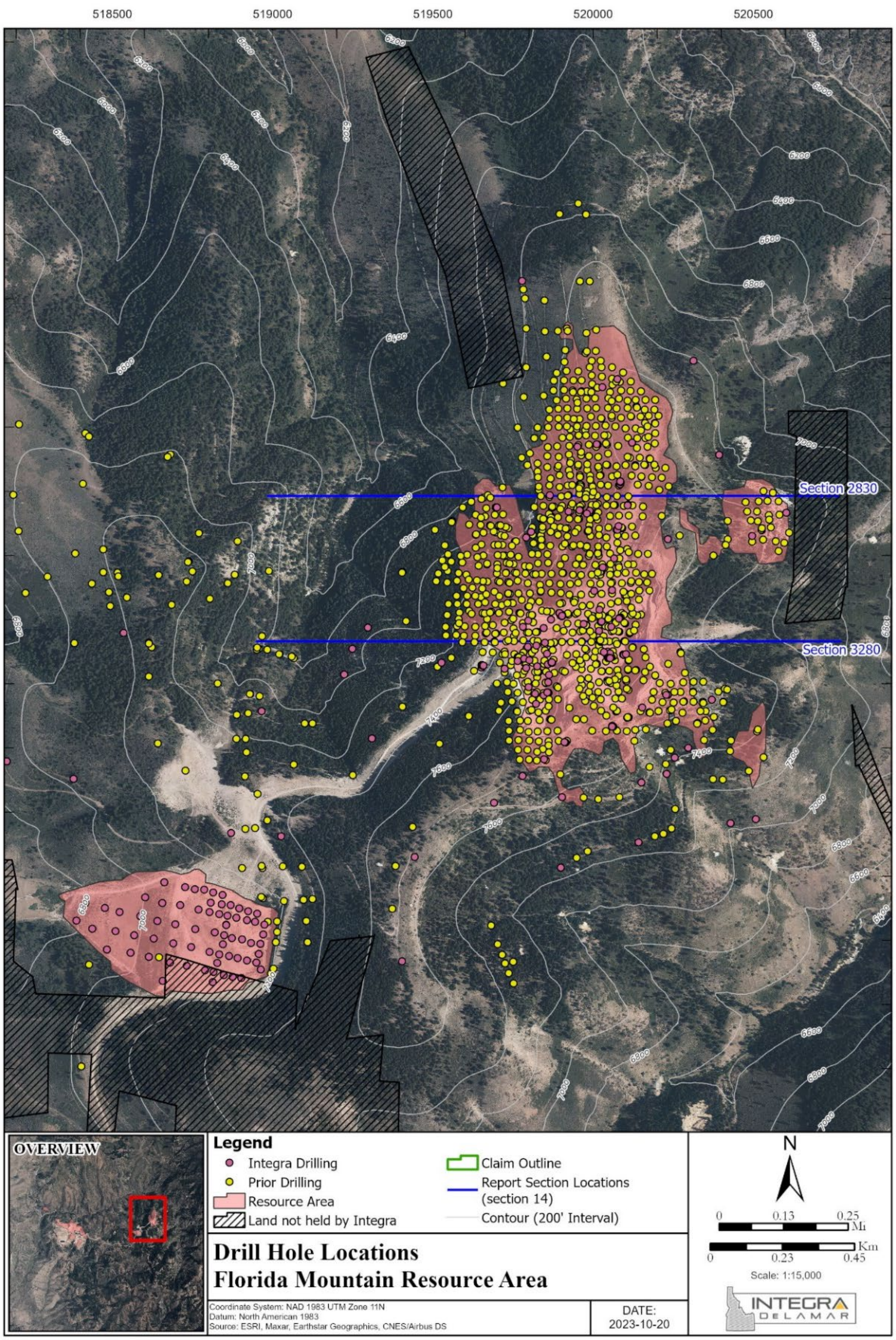


Figure 10-2 Map of Florida Mountain Area Drill Holes



10.2.3 Sidney Mining 1972

Sidney Mining drilled eight core holes in the Sommercamp and North DeLamar zones in 1972. RESPEC 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 (215,554.5 feet), 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 (600-foot) vertical RC hole.

Earth Resources drilled an additional 70 vertical holes of unknown type during this period, for a total of 5,202 meters (17,067 feet).

10.2.5 NERCO 1985 - 1992

Available records show NERCO drilled 691 holes from 1985 through 1992. These include 351 RC holes for a total of 37,093 meters (17,067 feet), seven conventional rotary holes drilled in 1986 for a total of 640 meters (2,099.7 feet), 36 core holes for 1,902 meters (6,240 feet), and 28,720 meters (94,225.7 feet) 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 tailing 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 RESPEC. 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) (14,734 feet) 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) (54,540.7 feet) 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 (518.4-foot) inclined RC hole was drilled in 1996, and two additional inclined RC holes, for a total of 91 meters (298.6 feet), 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 (11,171 feet) in 45 vertical rotary holes in the Florida Mountain area. This drilling was done by Eklund Drilling of Elko, Nevada, using 13.34-centimeter (5.25-inch) 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 (10-foot) intervals, but RESPEC 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 (1,899.6 feet) 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 (10-foot), but RESPEC 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 (2,135.8 feet). Eklund Drilling and D. Allen Drilling were the contractors. A Midway and Ingersoll Rand TH100 drill were used, respectively, with 13.34-centimeter (5.25-inch) diameter hammer bits. Samples were collected over 1.524-meter (5-foot) intervals, but RESPEC 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 (14,724.4 feet), were inclined HQ-diameter (63 millimeter) (2.5-inch) core holes, with the remainder drilled by RC methods (11,729 meters) (38,481 feet). Twenty-eight of these RC holes were drilled vertically. Incomplete records show that 5 water wells, for a total of 475 meters (1,558.4 feet), were also drilled in 1988. At least one, and possibly all, of the water wells were drilled with a CP650 drill operated by "Allberry". The authors are 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 (24,255.3 feet) 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. RESPEC 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 (32,483.6 feet) 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 (16,099 feet) in 49 holes. In 1997, a total of 4,658 meters (15,282 feet) 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 (5-foot) intervals, but RESPEC is unaware of any other specific drilling and sampling procedures and methods.

10.4 Integra Drilling 2018 -2023

Integra's drilling through April 2023 is summarized in Table 10-3.

10.4.1 DeLamar Area Drilling 2018 - 2022

A total of 47,828 meters (156,915 feet) were drilled in 199 holes in various parts of the DeLamar area in 2018 through 2022. Approximately 44% of the holes and 49% of the meters were drilled with RC or Rotary 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 five of the 2020 DeLamar area holes was vertical, with the others inclined at angles of -45° to -85°.

The RC drilling was conducted by Boart Longyear of Elko, Nevada in 2018, 2019 and 2022 using an MPD 1500 track-mounted drill and in 2021 by National EWP using what the drill logs refer to as a "National 176" rig. Bit diameters varied from 12.065 centimeters to 15.558 centimeters (4.75 to 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 (10 pounds) to 9.07 kilograms (20 pounds) 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 2019,2020 and 2021 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 2021 and 2022 core drilling at DeLamar was conducted by National EWP of Elko, Nevada and Tonatec Exploration of South Jordan, Utah using a track mounted LF90 core rig. PQ- and lesser HQ-size core was recovered with wireline methods and triple-tube coring.

Table 10-3 Integra Drilling Summary

Area/Target	Year	RC Holes	RC Meters	RC-Center Return Holes	RC-Center Return Meters	Core Holes	Core Meters	PC-Core Holes	PC-Core Meters	Rotary Holes	Rotary Meters	Sonic Holes	Sonic Meters	Total Holes	Total Meters
DeLamar															
Glen Silver	2018	3	1,018			6	1,433							9	2,451
Henrietta	2018	5	1,228											5	1,228
Milestone	2018	6	1,218											6	1,218
North Wahl	2018							1	201					1	201
Ohio	2018	7	2,960			4	919	2	383					13	4,262
Sommercamp	2018					2	367	8	1,799					10	2,167
Sullivan Gulch	2018	14	4,913			6	2,309							20	7,222
Sullivan Knob	2018	3	1,061											3	1,061
Town Road	2018	2	652											2	652
Glen Silver	2019					7	1,345							7	1,345
Ohio	2019					1	104							1	104
Sommercamp	2019					3	467							3	467
Sullivan Gulch	2019	16	7,285			5	1,964							21	9,249
Glen Silver	2020					3	336			2	94			5	430
Henrietta	2020					5	1,084			1	21			6	1,105
Milestone	2020					3	436			2	104			5	539
North Wahl	2020					7	465							7	465
Ohio	2020					5	818							5	818
Sommercamp	2020					2	221							2	221
Sullivan Gulch	2020					4	533			2	220			6	753
Heap Leach Facility	2021	3	229											3	229
Henrietta	2021	3	594			11	2,951							14	3,546
Milestone	2021	10	1,129							1	44			11	1,173
North Wahl	2021					6	1,594							6	1,594
Ohio	2021					1	255							1	255
Sullivan Gulch	2021					3	942			2	139			5	1,081
Sullivan Knob	2021									2	60			2	60
Town Road	2021									2	109			2	109
Glen Silver	2022					6	422							6	422
Henrietta	2022	2	152											2	152
North Delamar Backfill	2022	15	850									25	1,162	40	2,012
Ohio	2022					4	854							4	854
Waste Dump 1	2022	3	133									20	919	23	1,052
Waste Dump 2	2022	13	514									10	404	23	917
Sullivan Gulch	2022					6	2,394							6	2,394
North Delamar Backfill	2023			14	963							16	1,122	30	2,085
Sommercamp	2023			10	300							29	749	39	1,049
Waste Dump 1	2023			21	1,087							19	473	40	1,560
Waste Dump 2	2023			24	689							19	622	43	1,311
DeLamar Total	2018-2023	105	23,936	69	3,039	100	22,213	11	2,383	14	792	138	5,450	437	57,813
Florida Mountain															
Florida Mtn	2018					10	2,949							10	2,949
Florida Mtn	2019					33	9,083							33	9,083
Florida Mtn	2020					28	9,093							28	9,093
Florida Mtn	2021					50	17,758							50	17,758
Blue Gulch	2021					2	408							2	408
Tip Top	2023			4	152							11	448	15	600
Jacobs Gulch	2023			8	396							60	1,606	68	2,003
Florida Mountain Total	2018-2023			12	549	123	39,289					71	2,054	206	41,892
All Integra Drilling		105	23,936	81	3,587	223	61,502	11	2,383	14	792	209	7,504	643	99,706

The 2018 through 2022 drill core was placed in plastic or wooden 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 daily.

10.4.2 Florida Mountain Area Drilling 2018 - 2021

In the Florida Mountain area, a total of 39,289 meters (128,902.5 feet) were drilled in 123 core holes (Table 10-3). These holes were inclined at angles of -45° to -75°. The drilling was performed by Major Drilling and Boart Longyear using LF90 track-mounted drills and Tonatec Exploration, National EWP and Falcon Drilling using a 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.

10.4.3 Integra 2022-2023 Stockpile Drilling

The Backfill and Stockpile Drilling program commenced on October 6, 2022 and concluded on April 30, 2023. A total of 12,588m (41,300') were drilled in 321 holes. The stockpiles drilled at DeLamar were the North DeLamar and Sommercamp backfill areas ("NDM-SC"), DeLamar waste dump 1 ("Stockpile #1), and DeLamar waste dump 2 ("Stockpile #2"). At Florida Mountain, the backfill of the Tip Top pit ("TT") and the Jacob's Gulch waste dump ("JG") were drilled. A mix of RC and Sonic drilling methods were used, with 38% of the overall footage completed with RC. One RC rig and three different sonic rigs were used. The RC drilling was completed by Boart Longyear using a Foremost MPD 1500. Sonic drilling was mostly completed by Earth Drilling, a subsidiary of Harris Drilling. Earth Drilling utilized a track-mounted TerraSonic Mini Rig and a track mounted Boart Longyear LS600. Twentynine holes, for a total of 547.1 meters (1,795') with an average hole depth of 18.9 m (61.9'), were completed by Boart Longyear using a track-mounted LS250 mini sonic rig. All Sonic drilling was completed with 6" outer diameter casing and both Earth rigs utilized a 4 5/8" bit, while Boart Sonic utilized a 3" bit. RC drilling was completed with 6 5/8" outer diameter casing and a 5 1/2" bit.

The RC drilling was completed with a casing advance method, also referred to as the symmetrix system. This method advances casing with and over the bit. This method serves to keep the loose material of the backfill from collapsing in and contaminating the sample. A center-return bit was utilized about 70% of the time, especially when recovery was becoming problematic.

Sonic drilling employs the use of high-frequency, resonant energy generated inside the Sonic head to advance a core barrel and casing into subsurface formations. During drilling, the resonant energy is transferred down the drill string to the bit face at various Sonic frequencies. Simultaneously rotating the drill string evenly distributes the energy and impact at the bit face. After the core barrel is in place, the casing is sonically advanced over the core barrel, protecting the bore hole's integrity. The core barrel is retrieved, and the sample is then placed into a plastic sleeve labeled with the hole name and depth on either end of the sleeve.

All holes were drilled vertically and ranged in depth from 4.6 m (15') to 120.4 m (395'), with an average depth of 39.2 m (128.7'). The drill spacing was nominally 60 meters, with select areas having 30 m infill to test variability and provide additional metallurgical samples.

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 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. 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 through 2022 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). For the 2022 and 2023 Backfill and Stockpile drilling program, collars were surveyed with a combination of the Bad Elf GPS and a Juniper Geode GNS2 which provides sub meter accuracy.

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, primarily in the case of inclined holes. It is therefore likely that deviations occurred in the historical drill holes at the DeLamar project, particularly at Florida Mountain, but, as is discussed below, this is not considered to be a material issue in the estimation of the current project resources.

Integra used a REFLEX EZ-GYRO EG0270 and EG0142 down-hole survey tool to measure down-hole deviation in the 2018 through early 2022 drill holes, in addition to using a REFLEX SPRINT-IQ in 2022. The instruments were 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.

Integra did not complete down-hole surveys of the 2022-2023 stockpile drilling program as all holes were drilled vertically.

10.7 Sample Quality and Down-Hole Contamination

Down-hole contamination is always a concern with holes drilled by rotary methods (RC or conventional). 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, although these concerns are limited at the DeLamar project due to the shallow depths and vertical orientation of the rotary holes, and the fact that a significant quantity of the rotary data was mined out during the historical mining operations.

Some of the drill-hole logs reviewed by RESPEC 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 therefore 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 RESPEC 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 RESPEC to have suspicious results, they were excluded from use in the resource estimation. Intervals noted as having poor recovery were also flagged and not used in the estimation of the project resources. Beyond the historical notations of possible contamination, RESPEC noted a few 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) (298.6 feet) for the mostly vertical holes in the DeLamar area and 123-meter (403.5-foot) median down-hole depths for the predominantly angled holes at Florida Mountain). Few historical drill holes at the DeLamar area intersected the water table, while none did at Florida Mountain. A few of the deeper Integra RC holes drilled at the DeLamar area, which penetrated to depths significantly below the water table, do have strong evidence of down-hole contamination, and these intervals were flagged and removed from use in estimation of the resources.

Sample recoveries from sonic drilling at the stockpile areas were often low. For example, the average recovery recorded by Integra’s loggers from 138 of the sonic drill holes drilled at various DeLamar stockpile areas was 65%. Even given the difficulties in accurately determining recovery from sonic samples, which Mr. Gustin witnessed during a site visit that included time at active sonic (and RC) rigs and in Integra’s on-site logging and sampling facilities, as well as the problematic nature of drilling unconsolidated materials generally, the sonic sample recovery was far less than ideal. However, comparisons of the sonic-sample gold and silver values to those of adjacent RC holes, completed by Mr. Gustin both visually during the detailed explicit modeling involved in the estimation of the stockpile resources (see Section 14) and by statistical analyses, Mr. Gustin found the precious metal grades of the sonic samples to be very consistent with those derived from RC samples. Mr. Gustin believes the sonic drill samples are of sufficient quality as used in this report. However, the difficult nature of drilling the stockpile materials by any technique should continue to be considered in any future resource classification.

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 80% of the historical holes were inclined, and the holes were generally slightly longer than those in the DeLamar area. In consideration of the fact that any potential future mining operation that would rely in part on the reliability of the historical drill data would entail open-pit methods, the potential inaccuracies in the locations of drill samples imparted by the lack of down-hole surveys is not considered to be a material issue.

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.

The overwhelming majority of sample intervals in the DeLamar and Florida Mountain databases have a down-hole length of 1.52 meters (5.0 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.

Beyond the sample-quality issues discussed in Section 10.7, which were either identified and the affected samples removed from use in the estimation of the project resources, or judged not to be material as in the case of sample recovery from sonic drilling, Mr. Gustin is unaware of any sampling or sample-recovery factors that materially impact the accuracy and reliability of the drill-hole data, and he believes that the drill samples are of sufficient quality for the purposes used in this report.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

This section summarizes all information known to Mr. Gustin 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 from historical records as cited or provided by Ms. Kim Richardson. Ms. Richardson of Jordan Valley, Oregon, 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 operation in 1997. Ms. Richardson's contributions to this section are derived from personal correspondence with Mr. Gustin, an internal mine memorandum by Richardson (1985), and a recent informal summary document compiled at the request of Mr. Gustin. In this section, conversions from metric to U.S. customary units of measure are limited to the first occurrence of that measurement.

11.1 Historical Sample Preparation and Security

Mr. Gustin is 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 (10 pounds). When received at the mine laboratory, the samples were dried and crushed to -10 mesh. Splits of 150 milliliter (9.15 cubic inch) volumes were then pulverized to pulps with 90% passing 100 mesh. At the date of Elkin's report, one-assay-ton (30-gram) (1.06-ounce) aliquots were taken from these pulps for assaying.

Mr. Gustin is 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 been 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 daily. 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 to 2022 core sample intervals were sawed 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 effective December 1, 2020, to the ISO/IEC Standard 17025:2017 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. RESPEC 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 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 knows 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 (0.705 ounces) 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 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 inquarts 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 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 inquart 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 (7.05-ounce) sub-sample that was pulverized to nominal 200 mesh pulp. Gold and silver were determined on 30-gram (1.06-ounce) 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.

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 (2.12-ounce) 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 (0.018-ounce) aliquots. Rock samples that assayed greater than 5 g Au/t were re-analyzed by fire-assay fusion of 30-gram (1.06-ounce) 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 (2.205-pound) 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 5 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, 2019 and 2020 core samples were prepared and assayed at AAL for gold, silver, and multi-elements using the identical methods used for Integra's RC samples.

RC and core samples from the 2021, 2022, and 2023 drilling were dried upon arrival at AAL's Reno facility. The dry samples were fine crushed to 70% passing 75 microns. A Jones Riffle split of one-kilogram (2.205-pound) material was then pulverized to 85% passing 76 microns. The one-kilogram pulps were assayed for gold, silver, and multi-elements using the same assay analyses as Integra's 2018, 2019 and 2020 samples.

Sonic samples from the 2022 and 2023 drilling were first crushed to -1-inch material and then prepared and assayed at AAL for gold, silver, and multi-elements using the identical preparation and analytical methods as Integra's 2021, 2022 and 2023 RC and core 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.

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 have not yet been compiled by Integra. The analytical 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 for sample preparation, where required, and assaying: 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. An evaluation of this 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 Elkin (1993) report indicates that repeat (check) assays were routinely run at the mine laboratory, which was confirmed by Ms. Richardson. 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 assay. Original sample pulps and splits from every fourteenth coarse sample were also resubmitted to the mine laboratory on a routine basis. Mr. Gustin has not found detailed documentation of these check analyses, and therefore could not independently evaluate the results. Elkin also stated that duplicate samples were not being sent to outside laboratories at the time of his report.

The mine lab also completed duplicate MIBK analyses and/or fire assays as a check on the lab's original 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 (1988) 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. Further documentation of this program, including the check-assay results, has not been found.

11.6.2 Integra

Coarse blank material commercially produced certified reference materials ("CRMs"), RC field duplicates, and coarse-reject (or preparation) duplicates were inserted into the drill-sample streams as part of Integra's quality assurance/ quality control procedures. The blank material consisted of coarse fragments of basalt, and a blank was inserted approximately every 10th sample. Commercial CRMs were inserted as pulps at a frequency of approximately every 10th sample. The lab was requested to prepare and analyze a coarse-reject duplicate for every 22nd primary sample analyzed during the sonic drilling program of 2022 and 2023.

<u>CRMs</u>. CRM pulps were inserted into the primary sample stream by Integra and analyzed along 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 precious-metal deposits are positively skewed. Samples outside of the three-standard-deviation limit 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.

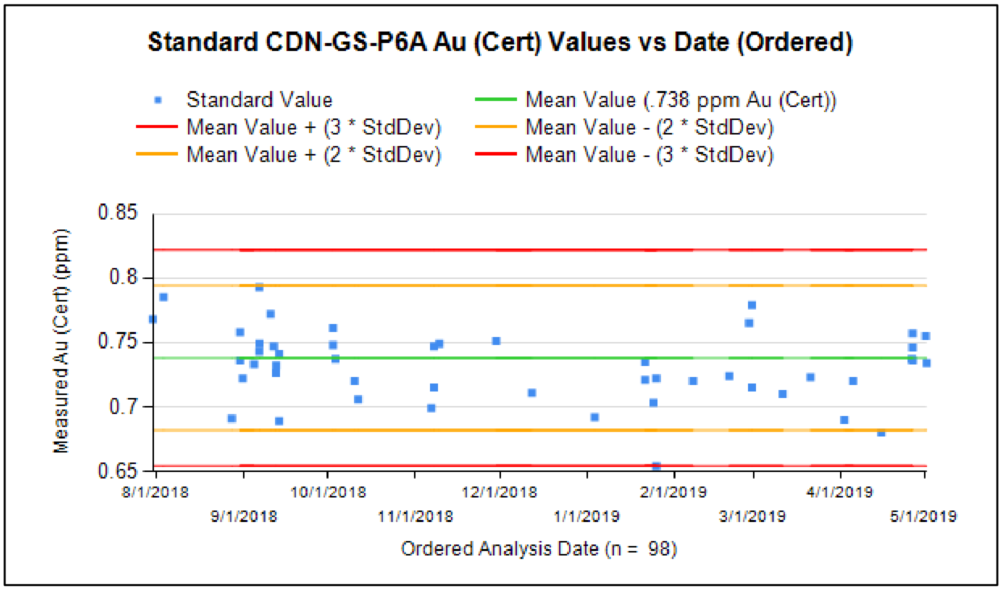
Table 11-1 lists the details of the CRMs that Integra has used at the DeLamar project for the drilling assays considered in this technical report.

Table 11-1 Integra Certified Reference Materials

CRM	Cert. Value (g Au/t)	2 SD (g Au/t)	Cert. Value (g Ag/t)	2 SD (g Ag/t)	No. of Analyses	Metals with Cert. Values	Years Used
CDN-CM-38	0.942	0.072	6	0.4	430	Au,Ag,Cu,Mo	2018-2021
CDN-CM-39	0.687	0.064	5.3	0.5	400	Au,Ag,Cu,Mo	2020-2021
CDN-CM-40	1.31	0.12	18	2	444	Au,Ag,Cu,Mo	2021-2022
CDN-CM-41	1.6	0.15	8	1	30	Au,Ag,Cu	2018-2019
CDN-GS-P5E	0.655	0.062	n/a	n/a	116	Au	2020-2021
CDN-GS-P5G	0.562	0.054	n/a	n/a	304	Au	2019-2022
CDN-GS-P6A	0.738	0.056	81	7	117	Au,Ag	2018-2019
CDN-GS-P8G	0.818	0.06	n/a	n/a	57	Au	2021-2022
CDN-GS-1P5Q	1.329	0.1	n/a	n/a	73	Au	2018-2020
CDN-GS-1V	1.02	0.098	71.7	5	73	Au,Ag	2019-2021
CDN-GS-1X	1.299	0.132	68.5	8.5	89	Au,Ag	2021-2022
CDN-GS-1Z	1.155	0.095	89.5	4.4	89	Au,Ag	2022-2022
KLEN 73915	1.08	0.03	n/a	n/a	243	Au	2018-2020
KLEN 74110	0.237	0.002	n/a	n/a	288	Au	2018-2020
KLEN 74383	4.93	0.1	47.6	4.8	102	Au,Ag	2018-2021
KLEN 74589	8.65 (non-grav)	0.36	4395	215	3	Au,Ag	2018-2019
	8.49 (grav)	0.22					
OREAS 211	0.768	0.054	0.214	0.038	177	Au,Ag	2022-2023
OREAS 236	1.85	0.12	0.478	0.115	52	Au,Ag	2022-2023
OREAS 251b	0.505	0.034	0.123	0.035	354	Au,Ag	2022-2023
OREAS 253b	1.24	0.07	0.348	0.049	40	Au,Ag	2022-2023
OXD108	0.414	0.024	n/a	n/a	34	Au	2018-2019
SL77	5.181	0.156	29.1	1.2	18	Au,Ag	2018-2019
SN74	8.981	0.222	51.5	1.5	148	Au,Ag	2018-2022
SP72	18.16	0.35	83	2.2	10	Au,Ag	2018-2019

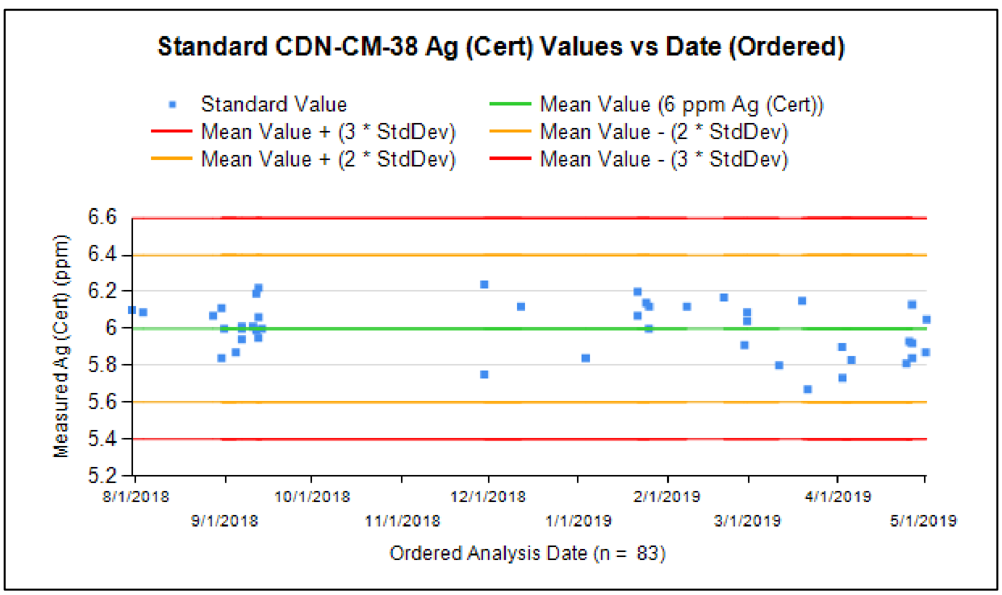
The AAL gold analyses of the CRMs inserted with the 2018-2019 drill samples met normal performance thresholds, with a moderate number of ‘failures’, although gold analyses of many either tended to have a low bias or clearly showed a low bias. Figure 11.1 shows a plot of the AAL gold analyses of CRM CDN-GS-P6A, 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 in 2018-2019.

Figure 11-1 CRM CDN-GS-P6A Gold Analyses – 2018-2019 Drill Programs



The AAL silver analyses of the eight CRMs used in the 2018-2019 drilling programs that have certified silver values in addition to certified gold values returned excellent results, with generally good precision and accuracy, leading to few failures and no bias, except for silver analyses of SN74, which show a high bias although without failures. Figure 11.2 shows typical results for AAL's silver analyses of the CRMs.

Figure 11-2 CRM SN74 Silver Analyses – 2018-2019 Drill Programs



A total of 837 CRMs were inserted into the drill-sample streams submitted to AAL from the beginning of 2020 through late August 2021, including some holes drilled after the resource database cutoff. All of these were analyzed for gold, while 574 were analyzed for silver as well. The results are summarized in

Figure 11.3 for gold and Figure 11.4 for silver, which show the results of all AAL analyses for all CRMs based on their standard deviations from the expected values.

Figure 11-3 All CRM Gold Analyses – 2020-2021 Drill Programs

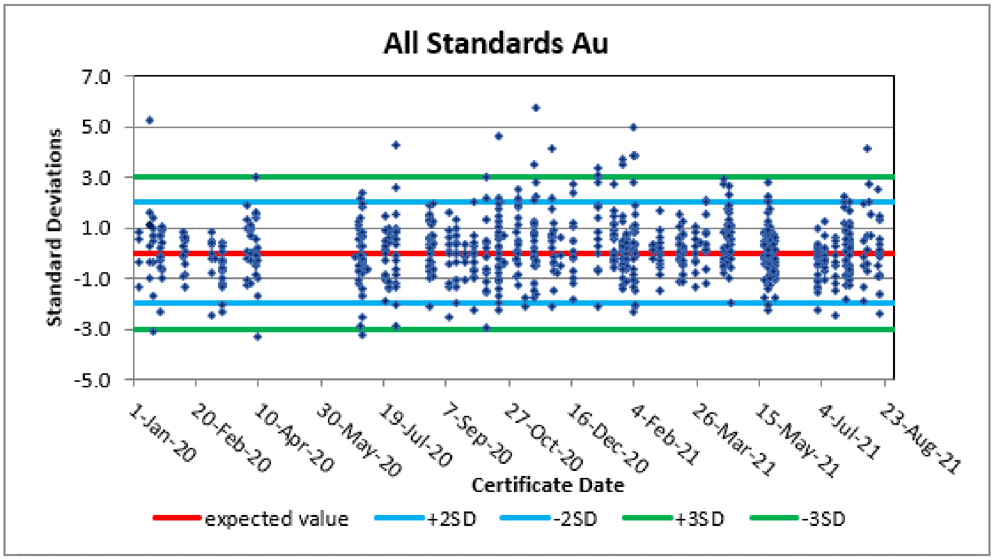
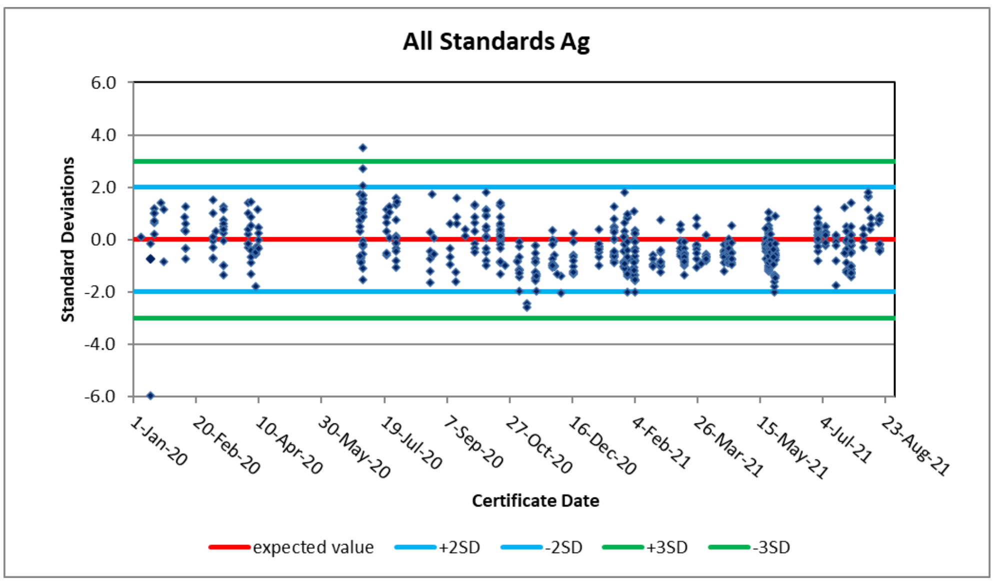


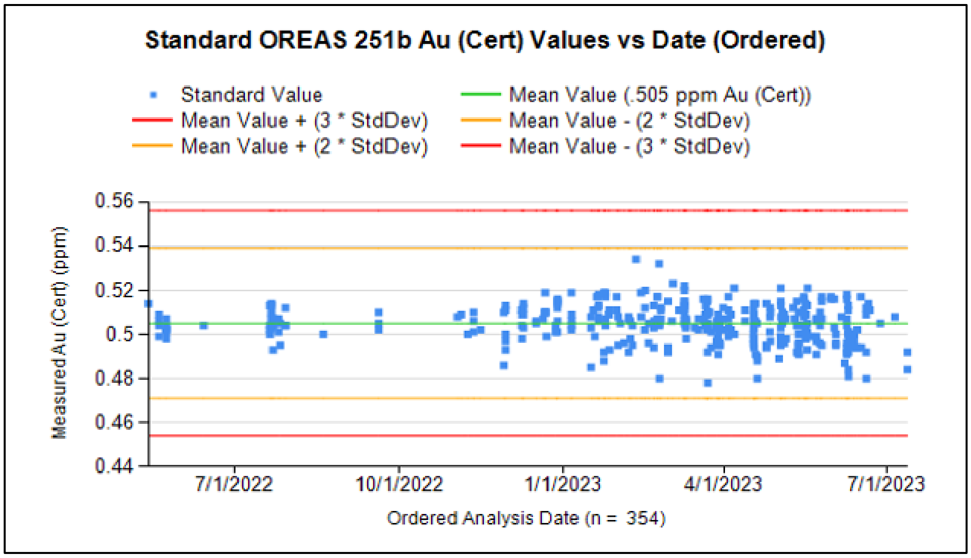
Figure 11-4 All CRM Silver Analyses – 2020-2021 Drill Programs



As shown in Figure 11.3 and Figure 11.4, the AAL analyses included 17 gold failures and two silver failures. Fourteen of the gold failures were to the high side, i.e., the AAL analyses exceeded the upper control limit of over three standard deviations; nine of these failures were from a single CRM (CDN-GS-P5G). The AAL gold and silver analyses of this CRM are biased high. Absent this problematic CRM, the failure rates for both gold and silver are less than 1%.

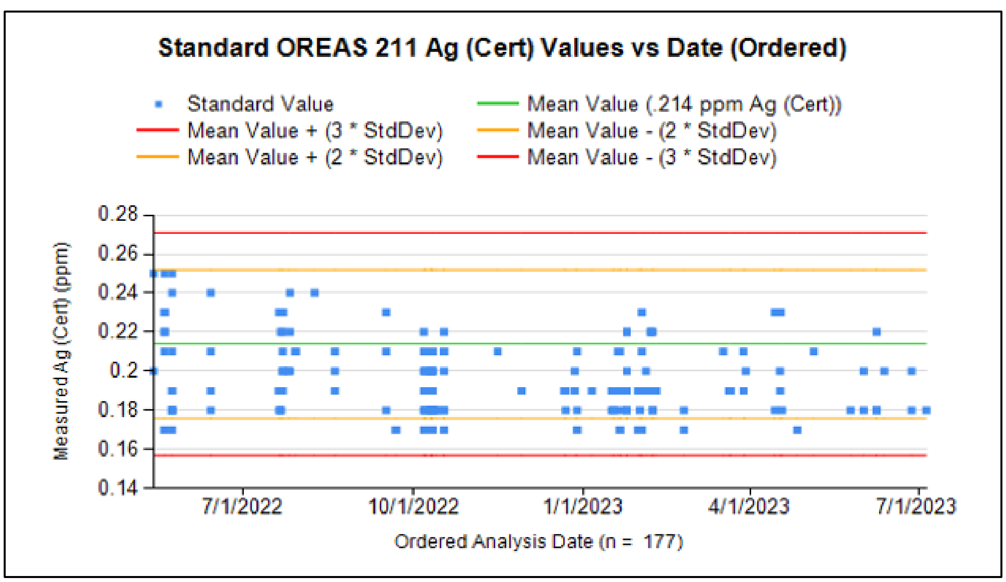
The AAL gold analyses of the four CRMs inserted with the 2022-2023 backfill and stockpile drill samples met normal performance thresholds, leading to few failures and no bias. Figure 11.5 shows a plot of the AAL gold analyses of CRM OREAS 251b, which has a certified value of 0.505 g Au/t. This CRM returned generally good precision and accuracy, and it is typical of the AAL gold analyses of most of the CRMs for the 2022-2023 backfill and stockpile drill program.

Figure 11-5 CRM OREAS 251b Gold Analyses – 2022-2023 Drill Programs



The AAL silver analyses of the four CRMs used in the 2022-2023 backfill and stockpile drilling programs that have certified silver values in addition to certified gold values met normal performance standards in terms of failures, but they display a tendency towards low bias. Figure 11.6 shows CRM OREAS 211 shows the low bias that is typical for AAL's silver analyses for 2022-2023.

Figure 11-6 CRM OREAS 211 Silver Analyses – 2022-2023 Drill Programs



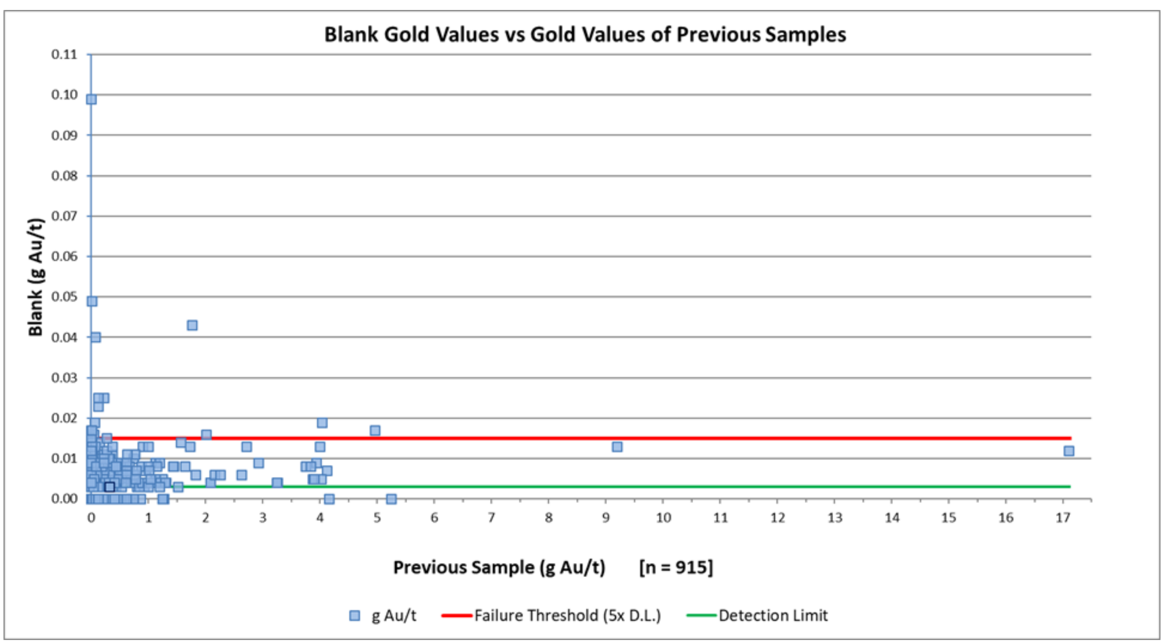
Coarse Blanks. Coarse blanks are samples of barren material that are used to detect possible contamination in the laboratory, which is most common during sample preparation stages. In order for analyses of blanks to be meaningful, they must be sufficiently coarse to require the same crushing and pulverizing 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 potential failures that should be subject to review and possible action. Figure 11.7 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.

Of the 915 AAL analyses of coarse blanks submitted with the 2018-2019 drilling programs, 14 exceeded the failure threshold, with blank assays 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.

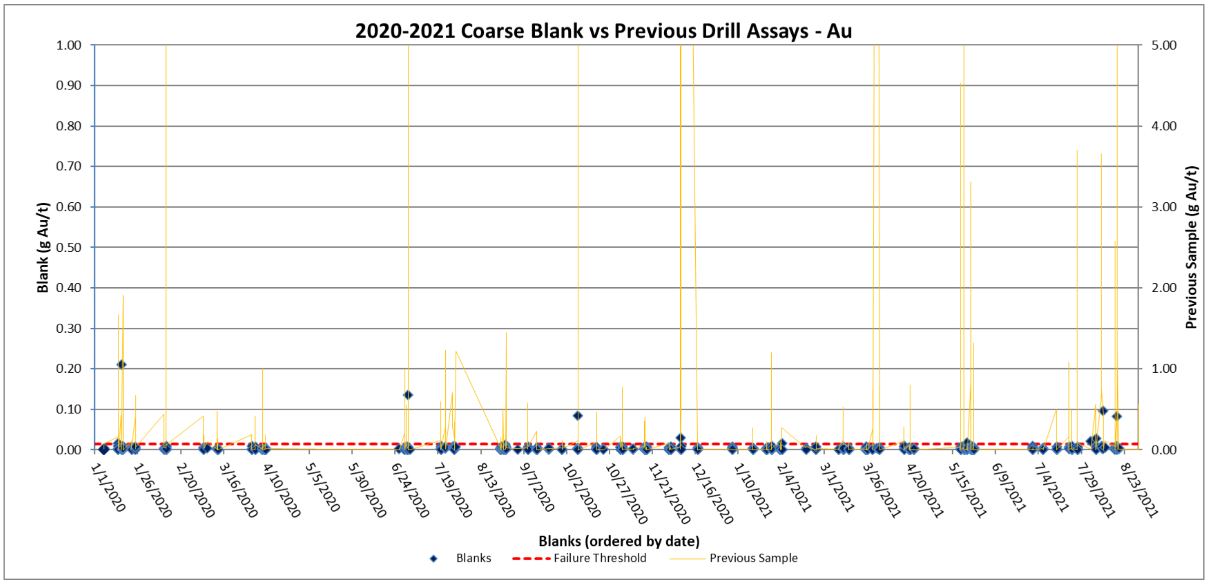
There are 889 AAL silver analyses of the coarse blanks. Using the reported detection limit of 0.020 g Ag/t, 93% of the AAL analyses of the blanks are technical failures. However, the highest value of the blank analyses is 4.86 g Ag/t, which is not of a magnitude that would be material to the project. Less than 3% of the AAL blank analyses are greater than 1.0 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.

Figure 11-7 Blank Gold Values vs. Gold Values of Previous Samples – 2018-2019 Drill Programs



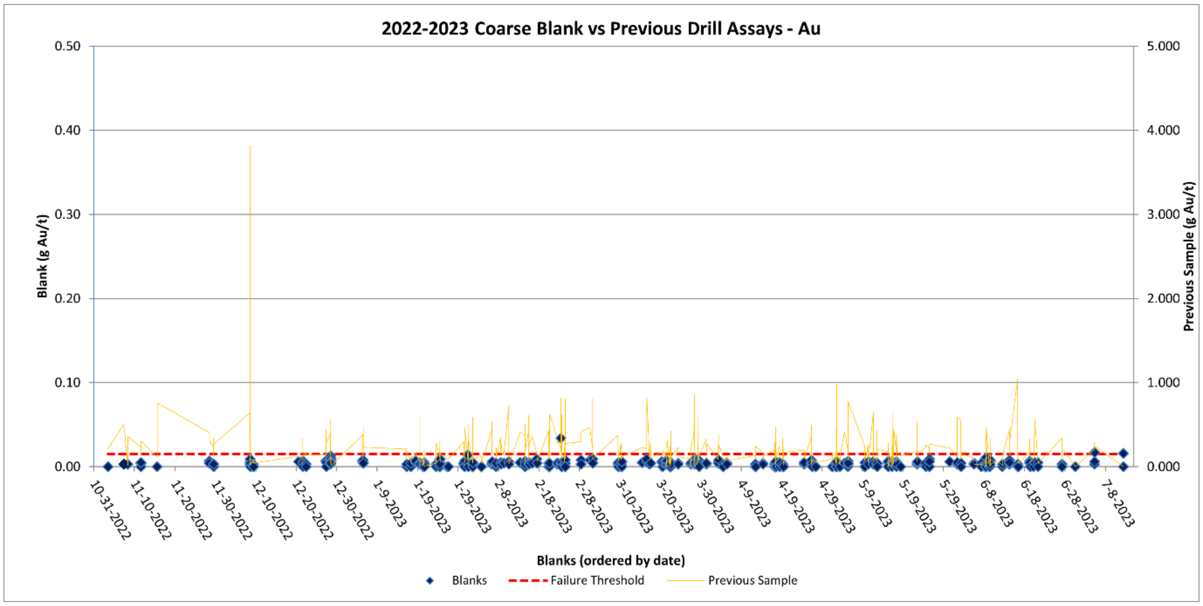
As part of the 2020-2021 drilling programs, 849 coarse blanks were inserted into the drill-sample streams and each blank was analyzed for both gold and silver. The AAL analyses produced 11 gold failures and 266 silver results that exceeded the putative threshold of 0.1 g Ag/t; only seven of the silver analyses exceeded 1.0 g Ag/t, with a high of 6.58 g Ag/t. Figure 11.8 shows the coarse blank gold assays compared to the preceding sample gold assays. Note that the gold failures generally correlate with higher-grade values from the preceding samples.

Figure 11-8 Blank Gold Values vs. Gold Values of Previous Samples – 2020-2021 Drill Programs



During the 2022-2023 stockpile drilling program, 412 coarse blanks were inserted into the drill-sample streams and each blank was analyzed for both gold and silver. The AAL analyses produced two gold failures and 29 silver results that exceeded the putative threshold of 0.1 g Ag/t; only one of the silver analyses exceeded 1.0 g Ag/t, with a high of 1.14 g Ag/t. Figure 11.9 shows the coarse blank gold assays compared to the preceding sample gold assays. Note that only one of the preceding samples had a gold value in excess of 0.1 g Au/t, which limits the usefulness of this dataset.

Figure 11-9 Blank Gold Values vs. Gold Values of Previous Samples – 2022-2023 Drill Programs



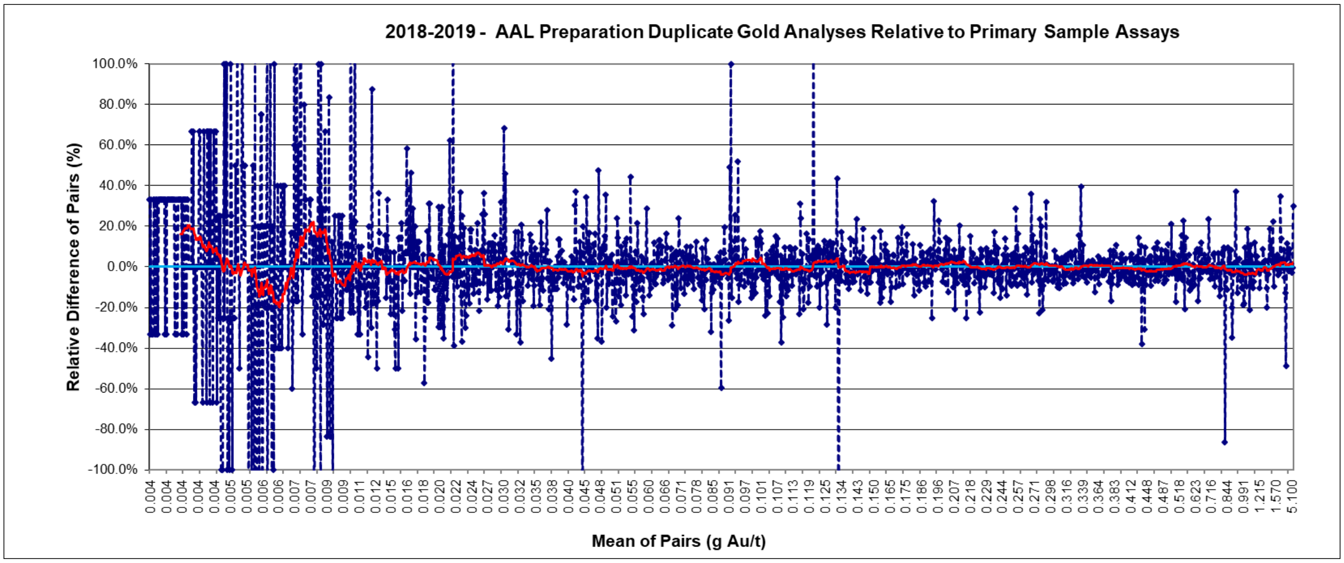
RC Field Duplicates. RC field duplicates are secondary splits of original 1.52-meter (five-foot) samples collected at the RC rig simultaneously with the primary sample splits. Field duplicates are used to evaluate the total variability introduced by subsampling, including at the drill rig and in the 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, all from the 2018-2019 drilling programs (no RC drilling was undertaken in the two deposit areas in 2020-2021). Figure 11.10 is a relative-difference graph that compares the RC duplicate data to the primary samples.

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 relative differences ("RDs") is +1%. The variability is well within an acceptable range, especially so for an epithermal deposit, with an average value of the RDs ("AVRD") of 14% that includes all data (no outliers removed). The AVRD decreases at higher grades (e.g., at a 0.2 g Au/t mean of the pairs cutoff, the average AVRD is 6%).

The silver field-duplicate data yields very similar results as the 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 mean of pair cutoff of 15 g Ag/t. As with gold, no outlier pairs were removed from these statistics.

Figure 11-10 RC Field Duplicate Gold Results Relative to Primary Sample Assays



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 is 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 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 is 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 quality assurance/quality control procedures and results are judged to be adequate.

12.0 DATA VERIFICATION

12.1 Drill-Hole Data Verification

The drill-hole databases that support the estimations of the DeLamar and Florida resources are comprised of historical data and subsequent information derived from Integra's 2018 through 2023 exploration programs. The historical portions of the databases were created by Mr. Gustin using the historical DeLamar mine digital database files for each of the two resource areas of the project.

The DeLamar area resource database includes information derived from 1,550 historical holes and 546 Integra holes. As the first step in verifying the historical data, a total of 235 of the historical holes were randomly chosen and their hole-collar coordinates, hole orientations, and gold and silver analytical information were compared to extensive, original, paper documentation in the possession of Integra. The database for the Florida Mountain area includes information from 1,107 historical drill holes, of which 169 were similarly checked, and 109 Integra holes that were all subjected to some level of checking. The results of this work, as well as other forms of data verification, are discussed below.

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 (59-foot) difference in elevation with the paper records, but the database elevation matches the project topography and was 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 (0.31 meters), or the truncation of surveyed decimals in the mine-site database. These discrepancies, which Mr. Gustin considers immaterial to the resource estimation, may reflect the perceived accuracy of the original drill-collar 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 were core holes, but no deviation data were found in the paper records for these holes. 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 approximately 25% of the Integra holes drilled at the DeLamar area were audited by comparing the information provided by Integra with original electronic files of the deviation 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 historical mine database and the updated survey information lie to the south of the current mineral resources. No discrepancies were found in the azimuth and dips of the audited holes.

In addition to the auditing of the database values, several holes at both the DeLamar and Florida Mountain areas were found whereby the hole collars were significantly above pre-mining topography and/or the assay results (and often logged lithologies) were not consistent with those of nearby holes, suggesting the hole locations were not properly represented in the historical databases. In the few cases where historical documentation could not resolve these issues, the holes were flagged in the databases and not used in subsequent resource estimations.

The drill-hole collar locations and down-hole surveys of approximately 25% of the 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 gold and silver 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 the database. Discrepancies between the RESPEC 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 were considered 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, and these data were added to the resource 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 eight gold and silver values were shifted down one sample interval (1.52 meters) (5.0 feet). 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 had 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. These discrepancies were corrected.

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.

During the auditing of the historical databases, certain gold analyses were found to be lacking in precision. These fire assays, primarily by independent commercial labs used by some of the earliest operators at the project, reported gold values in increments of 0.005 oz/ton (0.17 g/t). This lack of precision is particularly problematic at grade ranges of potential mining cutoffs, and especially so for low-cost open-pit operations such as is envisioned for the DeLamar project. A total of 11,197 DeLamar area gold analyses and 888 Florida Mountain area analyses were identified as being low precision; these sample intervals were flagged in the databases and were not used in the estimation of the project resources.

The following discussion summarizes statistical analyses of various duplicate datasets Mr. Gustin compiled from the historical mine-site databases following the verification and related corrective work discussed above.

The mine databases have up to three mine-lab analyses for certain sample intervals, although the nature of the material re-assayed (*e.g.*, pulps, coarse rejects, field duplicates) is not known. All available duplicate mine-lab analyses were compiled by Mr. Gustin and the duplicate datasets were evaluated.

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 were not below the detection limit, were examined. 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. In this and subsequent graphs, metal grades are shown in ounces-per-ton, honoring the units of the original analyses.

Figure 12-1 Repeat Mine Lab Silver Assays Relative to Original Mine Lab Assays

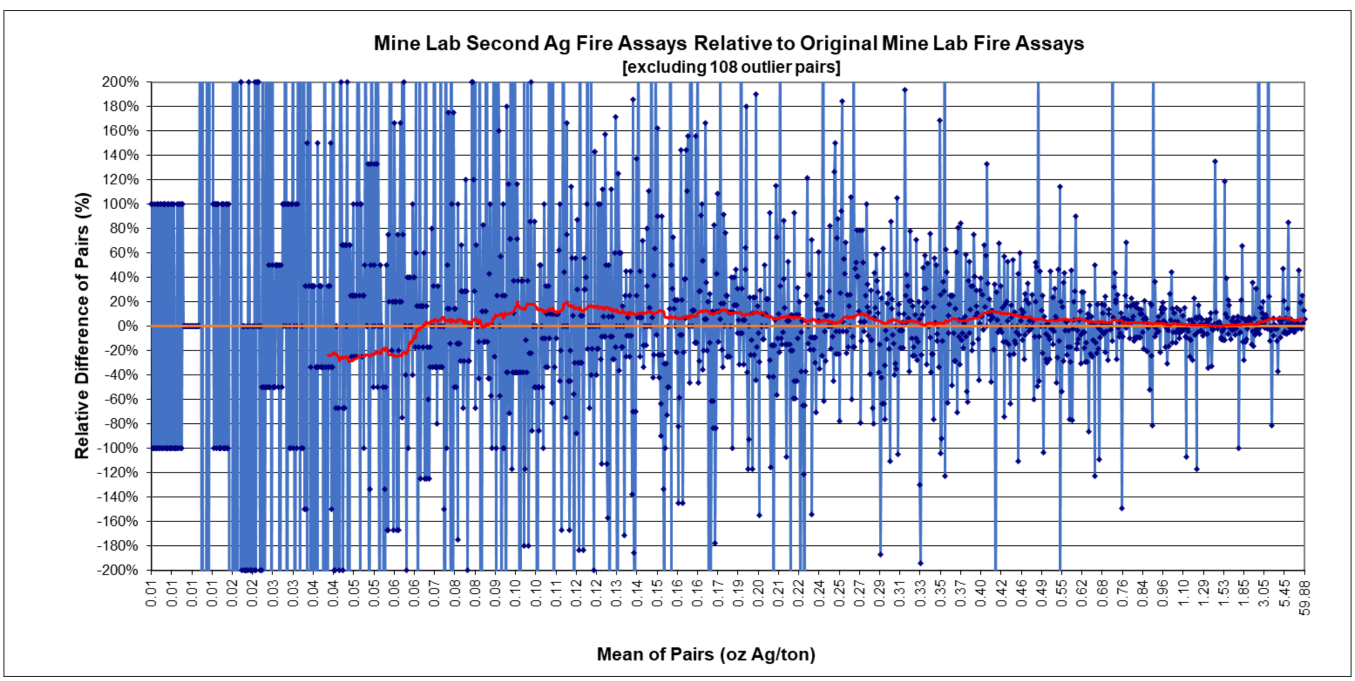


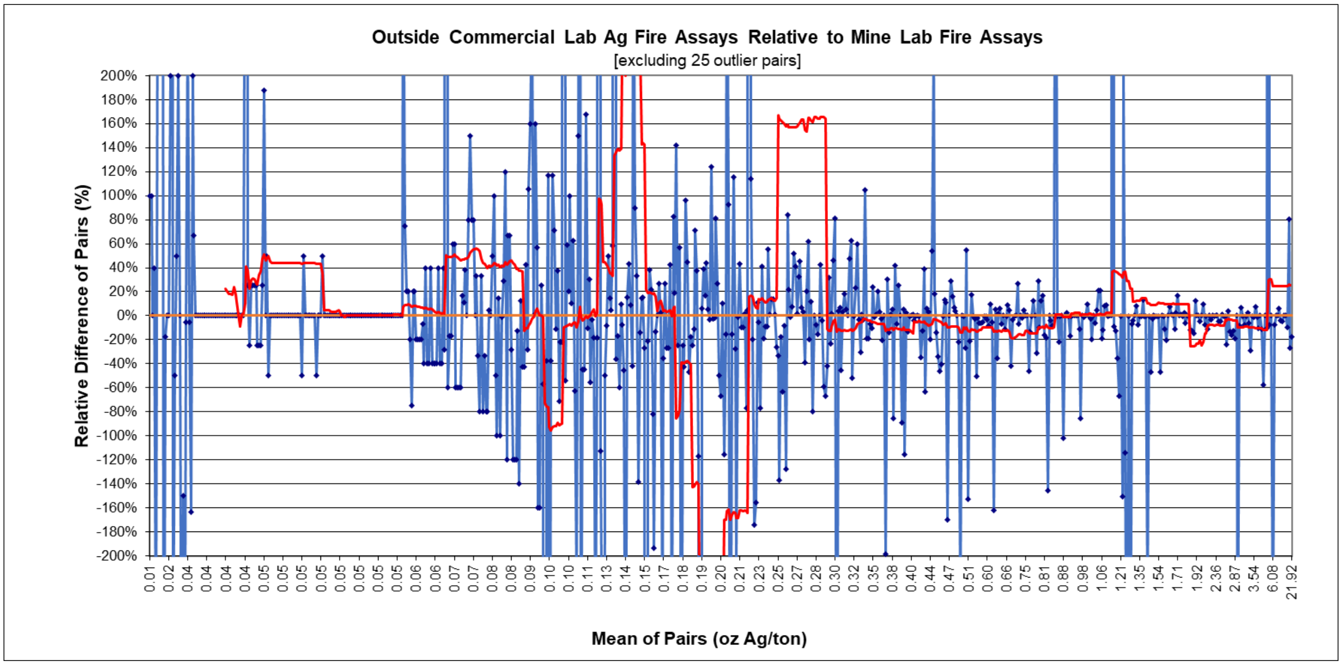
Figure 12.1 suggests a high bias of low magnitude in the duplicate silver results relative to the original assays over most of the grade range of the data. The mean of duplicate analyses is 0.613 oz Ag/ton (21.0 g Ag/t), which is 4% higher than that of the original results (0.588 oz Ag/ton; 20.2 g Ag/t), 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 wary 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%, suggesting that the duplicate analyses were not completed on original-sample pulps. 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 the high bias in the duplicates relative to the original analyses is present in what is a relatively low-grade silver dataset, and the magnitude of the high bias over much of the grade range is low.

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.

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 (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 important to note the relatively 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.

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 for this limited dataset 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).

Figure 12-2 Outside Lab Silver Assays Relative to Original Mine Lab Assays



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 documented the 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 for all mine-site purposes including reconciliations and historical estimations of resources and reserves. The FFAU and FFAG values prioritized fire assays completed by the mine site or outside laboratories over mine-lab AA analyses. The factored AA silver values were included in the FFAG field, while the original, unfactored AA silver analyses were also retained in the mine-site databases.

Mine lab AA silver analyses were reported to have been systematically low. Figure 12.3 compares data from 4,378 pairs of mine-lab fire assays and mine-lab AA analyses. A clear systematic bias is evident, whereby the AA analyses are lower than the paired fire assays, which is consistent with the mine staff’s observations. The mine site attributed this to incomplete digestions of silver minerals in the AA analyses. In an attempt to account for the digestion problem, the mine lab 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. The mean of the AA analyses is 22% lower than the fire-assay mean for all data, as well as at several MOP cutoffs inspected. The average RD also is more-or-less constant at approximately -30% for all cutoffs examined. No original or factored 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

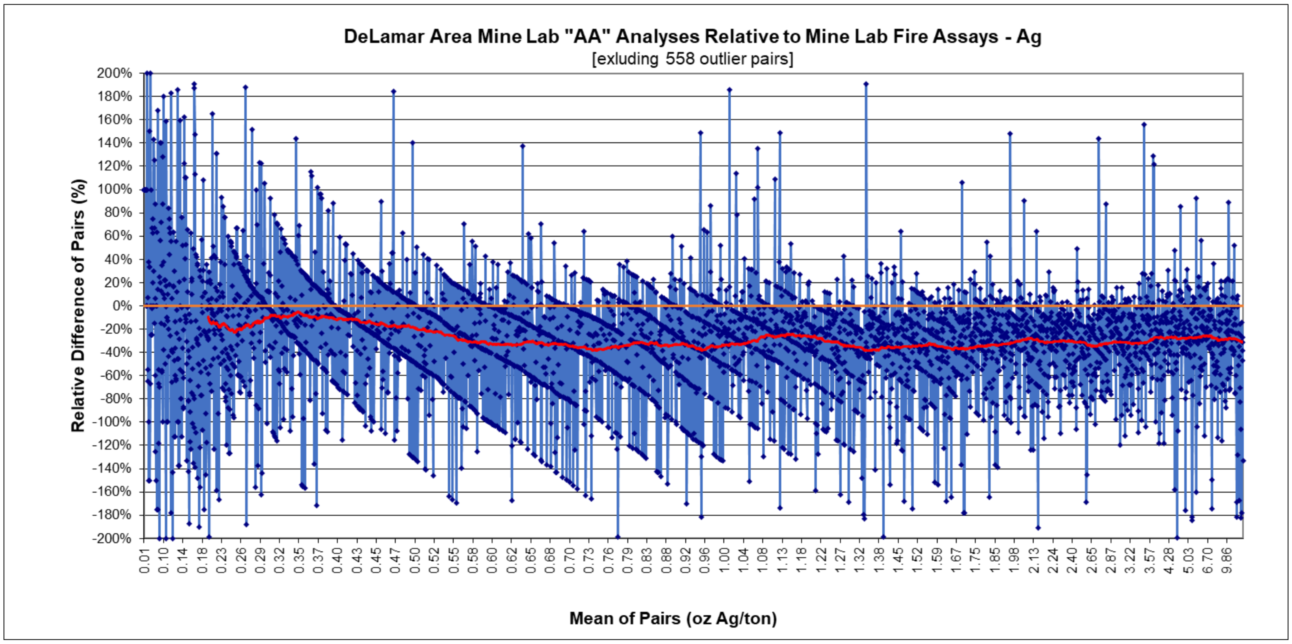
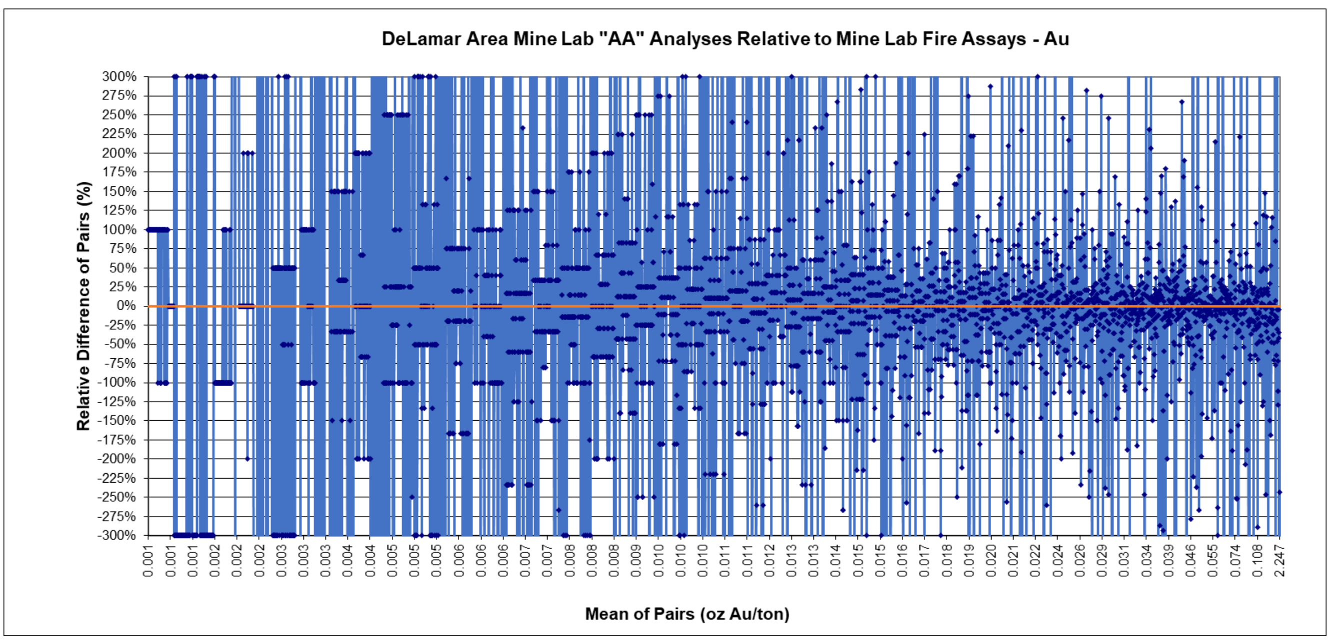


Figure 12.4 compares mine-lab gold AA analyses with mine-lab fire assays of samples from the same intervals.

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 AA gold values may be understating the actual grades of the samples. The AA gold values in sample intervals for which no fire assay gold analyses are available, which represent 29% of the historical gold assays in the DeLamar resource database and 24% in the Florida Mountain database, were accepted for use in the estimations of the current resources. All factored mine-lab AA silver values were removed from the resource databases. Unfactored mine-lab AA silver analyses that remain in the databases due to the lack of fire assays for those sample intervals were flagged and not used in the resource estimations, as these analyses demonstrably understate silver grades. In contrast, Mr. Gustin's analysis of mine-lab AA gold analyses found that they agree well with paired fire assay data up to a grade of 0.1 oz Au/ton (0.343 g Au/t), but at higher grades the AA gold analyses tend to be lower than the paired fire assays. While fire assays were prioritized over the AA gold analyses in the resource databases, the AA analyses are used for sample intervals lacking fire assays.

Integra Assays. Integra provided a complete assay compilation for all holes drilled in 2018 through April 2023. The sample numbers in these files were then linked by Mr. Gustin 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 a certain analytical method was chosen when multiple methods were available, and the method chosen by RESPEC differed from that in the Integra compilation.

12.1.3 Integra Data Verification

In addition to RESPEC's checking of historical data using historical records, Integra independently verified the accuracy of the 'from', 'to', and gold and silver assay values of every 10th sample interval, using RESPEC's audited database (see Section 9.4). The very few discrepancies found by Integra were then corrected in the resource databases, and this work also led to further checking of the surrounding sample intervals.

At Mr. Gustin's request, Integra also compiled data relevant to sample quality from historical drill logs and logged oxidation state using historical chipboards stored at the mine site.

12.2 Additional Data Verification

In addition to the more structured verification procedures discussed above, extensive verification of the project data, with an emphasis on the historical data, was undertaken throughout the process of the resource modeling. The careful work involved in the explicit modeling of the gold and silver mineralization within the context of the project geology 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 Mr. Gustin'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 holes' locations must be inaccurate, and these holes were therefore excluded from use in the resource estimation. Many individual historical assays, as well as assays within 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 important component to the verification of the historical data. Integra's ongoing drilling programs led to repeated updates of the resource databases. After each batch of new Integra drill data was added, the data were compared to the existing gold, silver, and lithologic modeling as it was being updated to reflect the new data. Where the Integra drill data penetrated areas at both the DeLamar and Florida Mountain areas that were previously modeled on the basis of historical data, the addition of the Integra data did not lead to material changes to the volume or grade of the gold and silver mineralization. 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.

To further verify the historical data in a more quantitative manner, the 2021 DeLamar and Florida Mountain resource models were compared to the 2019 models prior to updating the new models with additional density data (which led to slightly higher densities as compared to the 2019 models). In order to further assure the comparisons were meaningful, the optimized pits used to constrain the 2019 resources were used to tabulate the 2021 models. The resulting tabulation of the DeLamar 2021 model, using the 2019 resource pits and cutoff parameters, yielded 2% more tonnes and essentially identical gold and silver grades as compared to the 2019 model. At Florida Mountain, the 2021 model has essentially identical tonnes at a gold grade that is 0.01 g/t lower than 2019 and a very similar silver grade. The DeLamar 2021 resource model incorporated data from an additional 50 holes (primarily core) drilled by Integra as compared to the 2019 modeling, while the 2021 Florida Mountain modeling was updated by incorporating the results from an additional 44 Integra core holes. The very close correspondence of the 2021 and 2019 resource models within the identical volumes of the 2019 resources pits demonstrates that the Integra data are consistent with the historical data, which supports the visual validations of the historical data discussed above.

Similarly, the current 2023 Florida Mountain resources, excluding the Keys area that was not part of the previous (2021) resources, potential heap-leach Measured and Indicated resources have increased by 3% as compared to the 2021 estimation, while the current Measured and Indicated resources potentially amenable to mill-related processing increased by 2%. At DeLamar, the current Measured and Indicated resources increased by 4% for the heap leach resources and decreased by 3% in the mill scenarios. These differences between the current and 2021 resources are within the limits of accuracy inherent in the estimation of resources and are therefore immaterial. While this comparison was prepared to analyze if the updated resources would lead to material changes to the reserves, it also once again provides confirmation of the historical data as well as the modeling of the resources.

12.3 Site Inspections

Mr. Gustin visited the project site on October 16 through 18, 2018, October 15, 2020, October 27, 2020, October 19, 2022, and January 13, 2023, accompanied by various members of the Integra technical team. All principal areas of mineralization at the DeLamar and Florida Mountain areas were visited in the field during these visits, as well as exploration areas both on the project (Town Road – Henrietta) and north of the project on other Integra-controlled lands. 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. Actively drilling RC and core rigs were also visited at various times, 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.

Mr. Dyer completed a site visit on October 27, 2020. Mr. Dyer verified existing facilities and general topography.

Mr. Welsh performed an initial site appraisal for Integra in late June of 2019. He was escorted around the site by Messrs. Tom Jordan and Tim Arnold of Integra. The principal sites visited had been previously identified as potential heap-leach pad sites between DeLamar and Florida Mountain. As Mr. Welsh travelled between Jordan Valley and the mine site, he viewed a potential tailings location approximately 2.5 miles west of the mine in an undeveloped meadow. Generally, the access around the property was good via pickup truck and several legacy mining and exploration roads were observed that were available for future site evaluation and characterization. The sites visited included the legacy mine pits at Florida Mountain, the reclaimed waste-rock storage facilities in Rich Gulch, the north facing ridges between Florida Mountain and DeLamar mine area, and the accessible areas around the operating water treatment plant.

In August 2020, Mr. Welsh toured potential tailings and heap-leach facility sites with Mr. Tim Gerkin, a geologist. On the afternoon of August 28, 2020 Mr. Welsh visited the Slaughterhouse Gulch drainage between Jordan Creek and the headwater area. This trip resulted in changing the recommended locations of the heap-leach facility and the tailing storage facility from the PEA due to geotechnical and perceived permitting constraints under then new Idaho Interim Regulations for ore processing with cyanide.

Mr. Welsh also attended a pre-bid on-site meeting for prospective consultants on October 27, 2020. A brief trip to an overlook was made to view the proposed locations of the heap-leach pads and process area.

Mr. Nopola also completed a site visit on September 24, 2020 and reviewed the general project site and existing pit slope conditions.

12.4 Metallurgical Data Verification

Mr. McPartland visited the DeLamar project site on January 17, 2019. Most of the metallurgical test data used for the resources and the PFS was from testing conducted under the direct supervision of Mr. McPartland at McClelland. Where subcontractors were used to generate metallurgical or mineralogical test data, Mr. McPartland reviewed the reports submitted by the subcontractors concerning that testing. Mr. McPartland also reviewed reports concerning historical metallurgical testing, where available, though that historical testing was used only as general background information for the project metallurgy. Mr. McPartland considers the metallurgical test data to be of sufficient quality to be used in this report.

12.5 Mining & Geotechnical Data Verification

Mr. Dyer completed a site visit on October 27, 2020 and verified current infrastructure and mined pits, tailings, and stockpile / waste rock dump conditions, as well as the sites for similar facilities as presented in the PFS, all of which he used to complete the PFS mine planning and economic evaluation.

Mr. Nopola has supervised geotechnical data collection and analysis for the geotechnical characterization for open pit operations and waste rock facilities. The data used includes historical data as well as laboratory rock-mechanics tests, visual observations of performance of existing highwalls and waste rock facilities, core photographs, geological maps and three-dimensional geological models, and high resolution unmanned aerial vehicle point-cloud data. Mr. Nopola has verified that the data used meets the standards required for geotechnical analysis.

12.6 Summary Statement

Mr. Gustin has undertaken extensive verification of the historical data, and he has also reviewed the results from similar verification efforts completed by Integra. This work has identified very few errors in the transcription of field and assay data into the historical mine-site drill-hole databases. In addition, the documentation of gold and silver analytical methods for each historical sample interval allowed Mr. Gustin to identify and remove historical assay data that has insufficient quality for use in the estimations of the current resources.

Explicit modeling of the gold and silver mineralization was the most critical component to the estimation of the project mineral resources. This 'hands-on' approach provided meaningful verification of the historical data, whereby Integra infill drill data were found to be consistent with the continuity, widths, and grades of the gold and silver mineralization as defined by the historical drilling. Comparisons of the estimated grades and tonnages of the DeLamar and Florida Mountain areas, with and without substantial input of Integra data, also yielded consistent results and thereby provided further verification of the historical data. Finally, it is important to note that the historical data served as the basis to construct and operate the long-lived and successful historical mining operations at the DeLamar project.

Integra provided Mr. Gustin with lithological and structural interpretations of the DeLamar and Florida Mountain areas. Integra's geological modeling has been continually refined since the project was acquired, with interpretations of the DeLamar area geology in particular evolving as more drilling, particularly core drilling, was completed. Integra's geological interpretations have been used by Mr. Gustin as the basis for each successive estimation of the project gold and silver resources. Based on the resource models Mr. Gustin has completed during Integra's involvement in the project, all of which entailed detailed, explicit modeling methods that were completed within the overall context of Integra's geological interpretations, Mr. Gustin believes the Integra geological models are of a quality that provides high-level support to the current resource modeling.

The authors of this section of this report 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 believe that the project data are acceptable as used in this report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section, prepared under the supervision of Mr. Jack S. McPartland, Senior Metallurgist with McClelland Laboratories, Inc. ("McClelland") in Sparks, NV, 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 were developed for the processing methods selected for the preceding NI 43-101 technical report titled "Technical Report and Preliminary Feasibility Study for the DeLamar and Florida Mountain Gold-Silver Project, Owyhee County, Idaho, USA" dated March 22, 2022 with an effective date of January 24, 2022 and are summarized here. 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 multiple parts, historical (pre-1990) testing and current (2018-2023) testing for Integra, which includes PEA testing, PFS testing and post-PFS testing. The current metallurgical testing forms the basis for the recovery and reagent estimates used in this report. The samples used for the current testing generally come from the current mineral resources and are believed to be reasonably representative of the material considered for processing in the PFS and the resource considered in this report. 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 current (2018-2023) metallurgical testing. In general, the oxidation class of the samples used for historical testing (oxide, mixed or non-oxide) was not identified. Multiple metallurgical testing programs were conducted during the 1970s and 1980s on samples from the DeLamar and Florida Mountain deposits.

13.1 DeLamar Area Production 1977 – 1992

13.1.1 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. 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) indicated mill recoveries during the first 15 years of mine operation averaged 96.2% for gold and 79.5% for silver.

13.1.2 Cyanide Heap Leaching 1987 – 1990

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

Mine records from Integra, 2017 indicated that 2,344,037 tons of material, with an average grade of 31.78 g/tonne Ag and 0.41 g/tonne Au were stacked on the heap. It was reported by Integra that 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 tailing facility. Estimated recoveries were reported to be relatively poor (41% Au and 8% Ag). 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 mixed materials.

13.2 Historical 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 of the work conducted is presented in numerous reports shown in Table 13.1. As discussed earlier in this report, details concerning the origin of the sample tested generally were incomplete. Although results from the historical testing are generally consistent with the Integra testing, described later in this chapter, the historical testing was not used for development of recovery and reagent consumption models developed by Integra.

Table 13-1 Historic Mineralogy and Metallurgical Testing, DeLamar and Florida Mountain

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 Integra 2018-2023 Metallurgical Testing

A multi-phase 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 was conducted in two major phases, with testing done in support of the 2019 PEA (referred to as PEA testing), followed by testing in support of the 2022 PFS (referred to as PFS testing) conducted from mid-2019 (after the PEA testing) through 2021. Results from PEA testing were summarized in multiple reports (McPartland, 2019a, 2019b and 2019c). Results from the PFS testing were discussed in multiple reports (McPartland, 2020a, 2020b, 2020c, 2021a, 2021b, 2021c, 2021d and Wickens 2020), which were summarized in a combined report (McPartland, 2022). Results from post-PFS testing conducted in late 2021 through early 2023 were discussed in multiple reports (McPartland, 2023a) and a memorandum (McPartland, 2023b).

Metallurgical testing was focused on five main areas: 1) heap leaching of DeLamar oxide and mixed materials, 2) mill processing of DeLamar non-oxide materials, 3) heap leaching of Florida Mountain oxide and mixed materials, 4) mill processing of Florida Mountain non-oxide material and 5) heap leaching of historical dump and backfill materials. Testing on the oxide, mixed, dump and backfill materials was focused on either 2-stage or 3-stage crushing, followed by conventional cyanide heap leaching. Mill processing of the non-oxide material was focused on production of a flotation concentrate from ground feed, followed by regrinding and cyanide leaching of the concentrate, for both deposits.

The samples used for testing generally were selected to represent the current mineral resources and reserves, 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. Samples tested have mostly been drill core composites (297), with lesser numbers of RC composites (94, primarily from DeLamar – Sullivan Gulch, non-oxide material) and bulk samples (4).

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 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.

13.3.1 DeLamar and Florida Mountain Samples

Samples are identified by oxidation classification, which is based on drill hole logging by the Integra geology staff. The names of two of the three oxidation classifications were changed after the PEA was completed. The three major oxidation classes are now identified as oxide (unchanged, previously called oxide), mixed (previously called transitional) and non-oxide (previously called un-oxide). Where samples used for testing were identified by the old oxidation classification names in the referenced reports, those are changed to the new names herein for clarity purposes. These were changes in naming convention only and did not result in otherwise reclassifying the oxidation class for individual composites.

Drill composites were prepared considering area, oxidation, depth, lithology, alteration, grade, and grade continuity. In general, variability composites were at least 6.1 meters (20 feet) in length. Larger composites (typically for column testing or detailed mill testing) generally included 3.0 meter (10 feet) of drill sample below cutoff grade as dilution on either end of the composited interval, as well as below cutoff grade material from within the composited interval. Interval cutoff grades generally used for metallurgical compositing were 0.2 g Au equivalent/tonne for oxide and mixed composites and 0.3 g Au equivalent/tonne for mill (non-oxide) composites.

13.3.2 DeLamar and Florida Mountain Mineralogy

13.3.2.1 Bulk Mineralogy and Textural Analysis

A total of 67 metallurgical samples from the McClelland PEA and PFS testing program were submitted to Vidence, Inc., in Burnaby, British Columbia for automated SEM scans to determine the mineralogy and texture of the materials. The results were summarized by Vidence (Enter, 2021). The samples included 20 from the Florida Mountain deposit, of which three were oxide, 10 were mixed and seven were non-oxide material type samples. The samples included 47 from the DeLamar deposit, of which 24 were oxide, 10 were mixed and 13 were non-oxide material type samples.

Florida Mountain samples were found to have moderately variable quartz and feldspar content with quartz ranging from 42% to 73% and feldspar ranging from 9.5% to 47%. Plagioclase was sporadically distributed varying from absent to 10%. Sulfide minerals, as expected, were most common in the non-oxidized mineralization with trace amounts present in the mixed mineralization. Pyrite was the most common sulfide identified, up to 2.2%, with trace amounts of chalcopyrite, up to 0.11%. Jarosite was the only sulfate identified and was most abundant in the mixed mineralization, up to 0.29%. The oxide mineralization was found to contain trace to no sulfides and only trace amounts of jarosite.

Clay species were found to be moderately diverse and locally abundant. Muscovite and illite were found to be the main clay species ranging up to 14% and 9.1%, respectively. Lesser amounts of kaolinite, up to 4.5%, and Al-silicate clays, up to 0.5%, were also identified. Chlorite was found sporadically, specifically in the non-oxidized mineralization, at abundances up to 13%. A strong correlation exists between lithology and clay mineral assemblage. Muscovite is most abundant in samples where plagioclase is absent or small amounts with moderate amounts of K-feldspar. Observed kaolinite correlated with the abundance of plagioclase. Both correlations were spread across all three oxidation classifications.

The main mineralization identified in the DeLamar deposit across all oxidation states was quartz rich, up to 95%, with feldspar ranging from absent to 23%. Plagioclase was not identified in any of the samples analyzed for the deposit. Pyrite was the primary sulfide mineral identified and was present in trace amounts in all areas of the mixed mineralization except for the DeLamar North region. Pyrite was also detected in trace amounts in the Glen Silver and Sommercamp oxide mineralization as well. Non-oxide mineralization was analyzed from Sullivan Gulch and Sommercamp. Pyrite was the main sulfide mineral with abundances up to 5.9%. Sphalerite, arsenopyrite, and chalcopyrite were also present in trace amounts. Alunite and jarosite were identified as sulfate phases present in the Glen Silver, South Wahl, and Ohio oxidized and mixed mineralization with abundances up to 3.1%.

Clay species were common throughout all the areas of the DeLamar deposit with varying amounts of muscovite, illite, and kaolinite identified. The Milestone samples exhibited significantly more total clay in the oxidized mineralization (30%) than the samples of mixed (3%) that were analyzed. The clays within the Milestone area exhibit signs of being potentially sensitive to water. South Wahl had the most intensive clay study with 15 samples analyzed. Samples exhibited an inverse relationship between illite and kaolinite concentration. Low kaolinite content was also correlated with high abundance of K-feldspar. The South Wahl oxide samples generally contained relatively high levels of clays, with illite content varying from 1.78% to 40.76% and kaolinite content varying from 4.11% to 40.76%. The South Wahl mixed material exhibited lower levels of kaolinite with a similar range of illite content. Further work would be required to confirm potential water sensitivity/swelling tendencies of the clays for the entirety of the DeLamar deposit.

13.3.2.2 DeLamar Non-Oxide Sample Gold and Silver Deportment Analysis

A mineralogical assessment was conducted by BV Minerals, in Richmond, British Columbia on five DeLamar non-oxide (identified during testing as non-oxidized) composites, including two from Glen Silver (composites designated 4307-192 and 4307-199) and three from Sullivan Gulch (composites 4307-012, 120 and 165), as part of the McClelland PFS testing program, to determine general mineralogy as well as gold and silver deportment, liberation characteristics and mineral associations. The results were summarized in BV Minerals (2020). Gold CN/FA ratios for these composites generally were low (12.6% - 37.1%), with the exception of Sullivan Gulch composite 4307-120 (Au CN/FA of 66.4%). Testing included QEMSCAN particle mineral analysis ("PMA") and trace mineral search ("TMS"). Gold deportment relied heavily on analysis of gravity concentrate produced from each composite because of the low (gold) grade nature of the samples.

The samples were described as presenting low degrees of sulfide mineralization (2.6% - 9.8% sulfide minerals). Pyrite/marcasite was the primary sulfide mineral. The composites also contained 0.1% to 0.7% arsenopyrite. Trace amounts of other sulfide minerals were also noted. Non-sulfide minerals included quartz, muscovite/illite, K-feldspar and kaolinite (clay), in order of mineral abundance.

All of the gold found in the five composites presented as native gold or electrum. Gold grain sizes ranged from 0.5 microns to 50 microns and averaged 0.8 microns to 11.8 microns in circular diameter. The majority of gold occurrences were <10 microns. Over half of the gold in 4307-012 (SG), 165 (SG) and 199 (GS) was finer than 2 microns and was mostly locked with pyrite and non-sulfide gangue. Relatively coarse (20 – 50 microns) gold was also observed in two of the Sullivan Gulch composites (4307-012 and 4307-120). These coarse grains are expected to be gravity recoverable. It was noted by the mineralogist that based on comparison between the grade of the samples evaluated and the amount of gold identified during the examination, it was possible to conclude that the samples also likely contain significant quantities of gold in solid solution with sulfide minerals ("invisible" refractory gold). Only 5% to 30% of the gold observed in the other three composites presented with exposed surfaces. Some difficulties in recovering gold from these composites were anticipated because of the relatively fine-grained gold observed (0.8 microns to 5.7 microns average).

Silver was present in the samples mainly as naumannite (Ag_2Se), pyrargyrite (Ag_3SbS_3), freibergite ($(Ag,Cu,Fe)_{12}(Sb,As)_4S_{13}$) and, in the case of the Glen Silver composites, canfieldite (Ag_8SnS_6). The relative abundance of these minerals varied significantly between composites. Silver grain sizes ranged from 0.5 microns to 20 microns and were most commonly present in the 10 microns to 20 microns size range. Silver liberation ranged from about 9% to 38% at the size studied (nominal 90 microns). Unliberated silver minerals were predominantly locked in pyrite and non-sulfide gangue. Between 50% and 95% of the silver in all composites but 4307-192 (GS) presented exposed surfaces. Silver recoveries greater than 50% were expected by leaching.

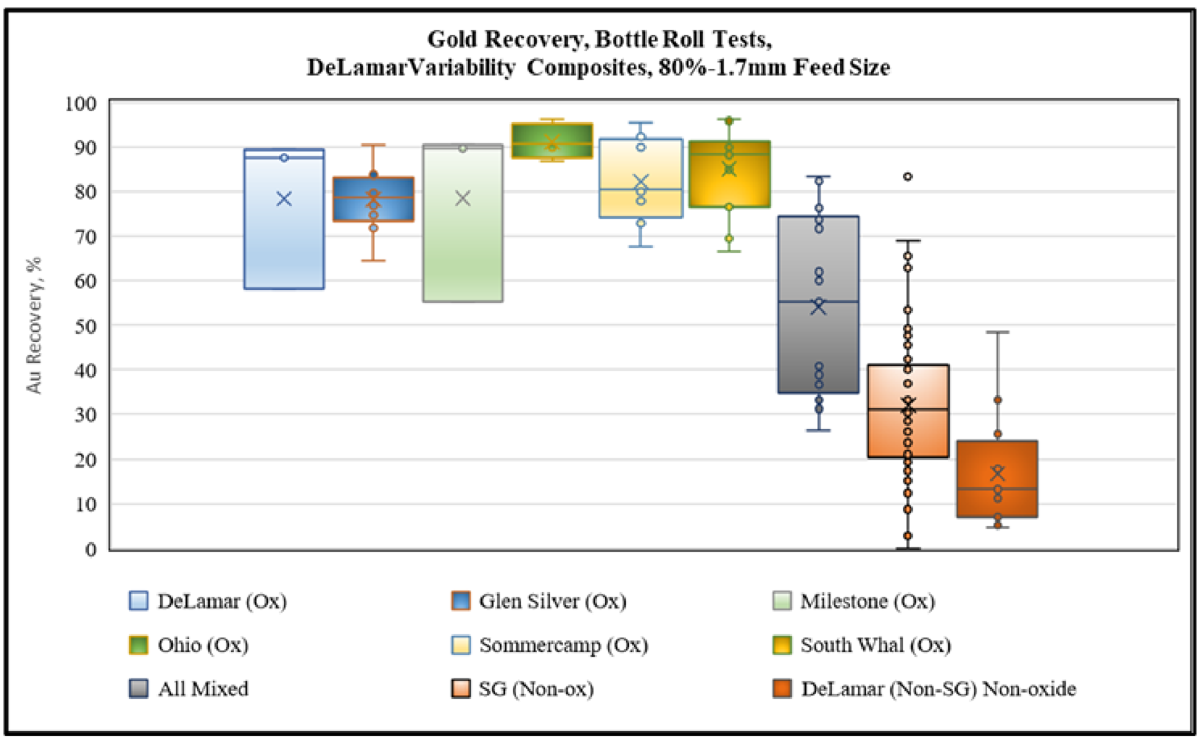
13.3.3 DeLamar and Florida Mountain Heap Leach Testing

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 mixed material types from both deposits can be processed by heap-leach cyanidation. These materials generally benefit from relatively fine crushing to maximize heap-leach recoveries and a feed size of 80% -12.7mm (0.5 inches) was selected as optimum.

13.3.3.1 DeLamar Bottle-Roll Testing

Bottle roll tests were conducted on each of the oxide (41) and mixed (17) variability composites, at an 80% -1.7 mm (10 mesh) feed size, to evaluate potential for heap leaching and material variability. Tests were also conducted on a total of 90 non-oxide drill composites. Gold recovery results are summarized in Figure 13.1.

Figure 13-1 Gold Recovery, Bottle-Roll Tests, DeLamar PFS Variability Composites



Variability bottle-roll test results showed that, in general, the oxide composites were readily amenable to agitated cyanidation treatment, at the 1.7 mm feed size. Gold recoveries from the 41 composites ranged from 55.3% to 96.3% and averaged 82.1% in four days of leaching. Silver recoveries obtained from the oxide composites were more variable and generally lower, compared to the gold recoveries. Silver recovery from the 41 composites ranged from 14.6% to 76.5% and averaged 39.2%, in four days of leaching.

A much smaller number (17) of mixed and blended (containing more than one oxidation type) composites were tested. Gold recoveries obtained in four days of leaching ranged from 26.5% to 83.3% and averaged 54.1%. Silver recoveries were more varied and on average were about 8% higher than obtained from the oxide composites. Silver recoveries from the mixed composites ranged from 20.0% to 87.7% and averaged 47.3%.

Bottle roll testing at the 1.7 mm feed size generally indicated that the non-oxide material will not be amenable to heap-leach processing. Gold and silver recoveries were low. Average gold recoveries from DeLamar (non-Sullivan Gulch) composites and from Sullivan Gulch composites were 16.8% and 32.1%, respectively. Average respective silver recoveries were 25.0% and 36.4%.

13.3.3.2 DeLamar Column Leach Testing

Column leach tests were conducted on each of 12 column test drill core composites and four bulk samples, representing DeLamar oxide and mixed material types, at an 80% -12.7 mm (0.5 inch) feed size, using 10 cm (4 inch) or 15 cm (6 inch) diameter by 3-meter (9.8-foot) high leaching columns. A 9.6 Lph/m^2 solution application rate and a cyanide concentration of 1.0 g NaCN/L were used. Column test duration generally was about 60 days. The core composite feeds were not agglomerated. Lime was added for pH control to the dry test charges before leaching. The bulk sample 80% -12.7 mm charges were agglomerated before column leaching using cement (5.0 – 10.0 kg/t). Comparative column-leach tests were conducted on the four bulk samples, at 100% -200 mm (8-inch) and 80% -50 mm (2-inch) feed sizes, to determine feed size sensitivity. Column leach tests were also initiated at a 100% -50 mm (-2-inch) feed size on three of the same 12 column test core composites tested at 12.7 mm (0.5-inch) and on a fourth column test core composite tested only at the -50 mm feed size. Results from those -50mm tests have not yet been formally reported and are not included in the results presented here. Comparative bottle-roll tests were conducted on each column test composite, at an 80% -1.7 mm (10 mesh) feed size to establish the relationship between recoveries from the two tests. Summary results from the bulk samples are presented in Table 13.2. Summary results from core composites are presented in Table 13.3.

Table 13-2 PEA Column-Leach and Bottle-Roll Tests, DeLamar and Glen Silver Bulk Samples

Sample	Oxidation Class	Test	Feed Size P_{80} (mm)	Leach Time, Days	Au Rec. %	Head Grade g Au/t	Ag Rec. %	Head Grade g Ag/t	NaCN Consumed, kg/t	Lime Added, kg/t
4307-A	Mixed	CLT	55	120	64.2	0.95	33.3	15	1.49	3.8
		CLT	50	99	68.0	0.97	43.8	16	1.30	6.7
		CLT	12.7	79	73.4	1.09	50.0	16	1.27	10.0*
		BRT	1.7	4	66.4	1.1	53.3	15	0.48	7.4
4307-B	Oxide	CLT	76	90	86.4	0.22	20.0	5	0.72	3.8
		CLT	50	63	88.0	0.25	20.0	5	0.63	3.8
		CLT	12.7	68	87.5	0.24	25.0	4	0.45	7.5*
		BRT	1.7	4	75.0	0.24	40.0	5	0.11	4.2
4307-C	Mixed	CLT	69	160	87.5	0.40	6.5	31	1.08	3.8
		CLT	50	83	90.0	0.40	13.8	29	1.01	3.1
		CLT	12.7	77	92.5	0.40	20.0	35	0.66	5.0*
		BRT	1.7	4	81.0	0.42	43.3	30	0.14	3.4
4307-D	Mixed	CLT	111	115	50.0	0.48	9.1	11	0.65	3.8
		CLT	50	138	67.2	0.64	10.0	10	1.27	3.3
		CLT	12.7	79	67.7	0.65	20.0	10	1.17	5.0*
		BRT	1.7	4	56.5	0.62	30.0	10	0.17	3.7

* Cement was used instead of lime and ore was agglomerated.

Note: CLT denotes column leach test and BRT denots bottle roll test.

Column testing showed that the Glen Silver bulk samples were amenable to heap-leach cyanidation treatment at the feed sizes evaluated. Gold recovery obtained from the oxide sample (4307-B) was not sensitive to feed size and ranged from 86.4% at the -200 mm feed size to 88.0% at the 80% -50 mm (-2-inch) feed size. Gold recovery rate was rapid and increased with decreasing feed size.

The Glen Silver mixed bulk samples were more sensitive to feed size. Gold recovery from sample 4307-A ("trans clay") increased from 64.2% at the -200 mm (-8-inch) feed size to 73.4% at the 80% -12.7 mm (-0.5-inch) feed size. Gold recovery from mixed sample 4307-C ("trans hard") improved from 87.5% at the -200 mm feed size to 92.5% at the 12.7 mm feed size. This sample contained only 0.06% sulfide sulfur and may be better classified as oxide material. Gold recovery from mixed sample 4307-D ("trans hard") improved from 50.0% at the -200 mm feed size to 67.7% at the 12.7 mm feed size. Gold recovery rates were slowest from the coarsest feeds, but generally were fairly rapid for the 50 mm and 12.7 mm feeds. Gold recovery rates from mixed sample 4307-D were very slow, and gold extraction was progressing at a significant rate from these feeds when leaching was ended.

Table 13-3 PFS Column-Leach Test and Bottle-Roll Test Results, DeLamar Core Composites

Composite	Area	Oxidation Class	Lith. Code	Sulfide Grade, % S	Test Type	Feed Size P_{80} (mm)	Time, Days	Au Rec. %	Head Grade g Au/t	Ag Rec. %	Head Grade g Ag/t	NaCN Consumed, kg/t	Lime Added, kg/t
4522-046	MS	Mix/Ox	Tpr	<0.01	CLT	12.5	131	47.8	0.67	15.8	19	1.59	1.1
					BRT	1.7	4	50.0	0.60	28.6	21	<0.07	1.2
4522-047	GS	Oxide	Tpr	0.05	CLT	12.5	53	72.2	0.36	12.5	16	1.46	2.1
					BRT	1.7	4	82.9	0.35	41.2	17	0.12	2.3
4522-048	GS	Oxide	Tpr	0.29	CLT	12.5	59	69.2	0.52	23.5	17	1.48	2.3
					BRT	1.7	4	75.6	0.45	41.2	17	0.18	2.5
4522-049	GS	Oxide	Tpr	0.16	CLT	12.5	55	71.8	0.39	16.7	18	1.05	2.6
					BRT	1.7	4	78.9	0.38	35.7	14	0.11	2.9
4522-050	GS	Mixed	Tql/Tpr	1.37	CLT	12.5	62	24.2	0.95	39.3	28	1.44	1.9
					BRT	1.7	4	32.6	0.86	53.8	26	0.39	1.9
4522-051	GS	Mixed	Tpr	0.07	CLT	12.5	62	75.9	1.45	19.1	89	1.41	1.8
					BRT	1.7	4	76.1	1.34	41.1	112	0.28	1.8
4522-052	SC	Oxide	Tpr	0.01	CLT	12.5	52	90.5	0.21	13.3	30	0.91	1.1
					BRT	1.7	4	94.1	0.17	27.6	29	<0.07	1.2
4522-053	DLMN	Oxide	Tql	0.01	CLT	12.5	52	81.3	0.16	8.3	36	0.94	1.5
					BRT	1.7	4	92.9	0.14	23.7	38	0.08	1.7
4522-054	Ohio	Oxide	Tbr	0.01	CLT	12.5	52	92.3	0.39	13.3	15	1.54	3.2
					BRT	1.7	4	92.5	0.40	27.3	11	0.18	3.6
4522-055	Ohio	Oxide	Tpr	0.01	CLT	12.5	55	80.0	0.15	18.8	16	1.46	2.1
					BRT	1.7	4	85.7	0.14	35.7	14	0.11	2.3
4522-071	SW	Oxide	Blend	0.01	CLT	12.5	53	67.6	0.34	27.8	18	1.12	2.5
					BRT	1.7	4	73.3	0.30	44.4	18	<0.07	2.8
4522-072	SW	Mixed	Blend	0.84	CLT	12.5	53	52.4	0.21	63.6	22	1.12	3.3
					BRT	1.7	4	47.8	0.23	56.0	25	0.33	3.3

Note: MS denotes Milestone, GS denotes Glen Silver, SC denotes Sommercamp, DLMN denotes DeLamar North, SW denotes SW.

The oxide composites were all amenable to simulated heap leaching, at an 80% -12.7mm (-0.5-inch) feed size. The mixed composites were more variable with respect to gold and silver recoveries obtained at an 80% -12.7 mm feed size, but generally were amenable to simulated heap leaching. Gold recoveries from the oxide core composites ranged from 67.6% to 93.2% and averaged 78.1%, in 53 to 59 days of leaching.

Column test silver recoveries obtained from the oxide composites ranged from 8.8% to 29.3% and averaged 16.7%. Silver extraction, and in some cases gold extraction, were progressing at a slow rate when leaching was terminated (<60 days). Silver recovery in particular would be expected to increase significantly with longer leaching cycles.

Column test gold recoveries from the mixed composites were lower and variable. Gold recovery obtained from the high sulfide (1.37% S) Glen Silver composite (4522-050) was very low (24.2%). The CN/FA ratio for this composite was also very low (39.4%) and the composite is thought to be better classified as non-oxide material. Column test gold recovery from the other Glen Silver mixed composite was substantially higher (75.9%). These conflicting data demonstrate the need for further refinement of the oxidation logging and modelling for the DeLamar deposit. Column test gold recoveries from the two other mixed ore type composites were 47.8% (Milestone Comp. 4522-046) and 52.4% (South Wahl Comp. 4522-072).

Column test silver recoveries from the four mixed composites were variable and ranged from 15.8% to 63.6%. As discussed below, silver extraction generally was incomplete when the tests were ended, and silver recovery tended to be higher for the samples with higher sulfide sulfur grades. Gold and, in particular, silver extractions were progressing at a slow rate when leaching was terminated for the mixed samples. Longer leaching cycles would be expected to improve gold recoveries incrementally and silver recoveries moderately.

Cyanide consumptions were moderate and ranged from 0.91 to 1.54 kg NaCN/t for the oxide composites and from 1.12 to 1.59 kg NaCN/t for the mixed composites.

None of the DeLamar samples displayed any solution percolation/permeability problems during column leaching. The bulk sample 12.7 mm (0.5-inch) column feeds were agglomerated with cement as a precautionary measure before leaching. Later testing indicated that those feeds did not require agglomeration pretreatment. None of the other DeLamar column feeds were agglomerated.

13.3.3.3 DeLamar Load/Permeability Testing

PEA and PFS Testing

Load/permeability (fixed-wall, saturated hydraulic conductivity tests; test procedure USBR 5600-89) were conducted on select samples from the PEA and PFS metallurgical testing. The tests were conducted to determine ore permeability under compressive loadings simulating planned commercial heap stack heights.

Load/permeability tests were conducted on the four bulk samples from the PEA testing at the 12.7 mm feed size. Fresh material was used for these tests. These samples were submitted to Geo-Logic Associates, in Sparks, Nevada, for load/permeability tests (test procedure USBR 5600-89). Results showed acceptable permeability under load without agglomeration pretreatment.

Select 12.7mm (0.5-inch) feed size column-leached residues (6) from the PFS column testing program were submitted to Geo-Logic Associates, in Sparks, Nevada, for load/permeability tests. Results from these tests indicated that, without agglomeration pretreatment, marginal to poor permeabilities were obtained at simulated heap stack heights greater than 30 to 80 meters (98.4 to 262.5 feet). These samples had variable, but generally low to moderate fines content (5% to 16% passing 75µm material).

After these results were reviewed, a weighted average composite of the 12.7 mm column residues was prepared for agglomeration testing. A series of bench-scale agglomeration tests were conducted on the composite to optimize cement binder addition. Results indicated a cement addition of approximately 3 to 5 kg/t was optimal. Based on results from the agglomeration testing, two agglomerated samples (using 3.0 kg/t and 5.0 kg/t cement) were prepared from the same column residue composite at McClelland and submitted to Geo-Logic in Sparks, NV, for load/permeability testing. Results indicated that for the material types displaying poor permeability characteristics, agglomeration using 3.0 kg/t cement should be sufficient to ensure adequate solution percolation characteristics for heap stack heights of up to about 80 meters (262.5 feet). For higher heap stack heights, a higher cement binder addition of 4.0 to 5.0 kg/t will likely be required.

Post PFS Testing

Considering the relatively poor results obtained without agglomeration, representative splits from the same six PFS testing column residues were submitted to a second Geo-Logic laboratory site (Grass Valley, CA) for replicate testing. Results from those tests indicated substantially higher hydraulic conductivities (1.6×10^{-3} cm/s or higher) at simulated heap stack heights of as high as 137 to 147 meters. These results indicated good potential for heap leaching at the planned heap stack heights, without agglomeration pretreatment, for most of the material tested.

Considering the disparity in results between the two sets of tests, two of the six PFS column residue samples were recreated by recrushing residues from the corresponding -50mm column leached residues. PSD analyses confirmed that the regenerated samples had essentially the same PSD as the original samples previously tested. Load-permeability tests were conducted on these samples, at two Geo-Logic Laboratories in Grass Valley, CA and in Reno, NV, without agglomeration pretreatment. Results showed much higher hydraulic conductivities ($>1 \times 10^{-2}$ cm/s) under compressive loading simulating heap stack heights of as high as 137 meters. Results from the tests conducted at the two laboratories agreed closely. These results indicated that the material represented by these samples would not require agglomeration pretreatment for heap leaching to heap stack heights of as much as 137 meters. These results call into question the reliability of the original hydraulic conductivity test results on non-agglomerated PFS column residue samples, particularly those displaying very poor permeability. Results from this work are presented in multiple reports (Plant 2023a, 2023b).

13.3.3.4 Florida Mountain Bottle-Roll Testing

Bottle roll tests were conducted on each of the oxide (28) and mixed (38) variability composites, at an 80% -1.7mm feed size, to evaluate potential for heap leaching and material variability. An additional 20 non-oxide composites were also tested using the same procedures. The tests were conducted using 4-day leach cycles with a cyanide concentration of 1.0 g NaCN/L. Summary results from bottle-roll tests are presented in Figure 13.2.

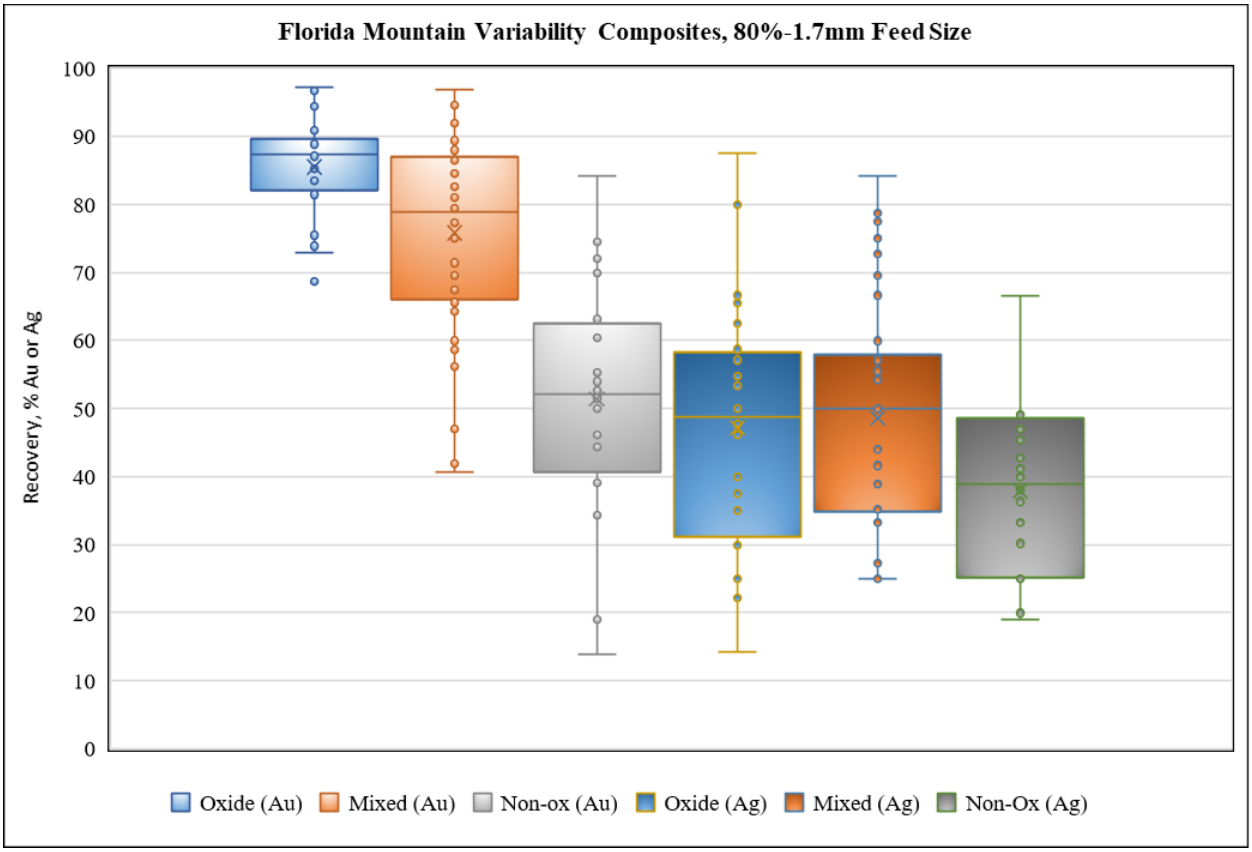
The oxide composites were readily amenable to agitated cyanidation treatment at the 1.7mm (10-mesh) feed size. Gold recoveries from the 28 composites ranged from 68.7% to 97.2% and averaged 85.5% in four days of leaching. Only five of the 28 composites tested gave gold recoveries of <80%.

Silver recoveries obtained from the oxide composites were highly variable and generally lower compared to the gold recoveries. Silver recovery from the 26 composites ranged from 14.3% to 87.5% and averaged 47.2% in four days of leaching. One-half of the composites gave silver recoveries of <45%.

A total of 38 mixed composites were tested. Overall, gold recoveries obtained in four days of leaching ranged from 40.6% to 97.0% and averaged 75.8%. About one-half (20 of 38) of the mixed composite gold recoveries were lower than 80%. Although gold recoveries from the mixed material were not strongly correlated to composite sulfide grade, the PEA (2018 drilling) mixed composites tended to have lower sulfide grades (0.08% average) and higher gold recoveries (81.4% average) compared to the PFS (2019 drilling) mixed composites (0.35% sulfide sulfur and 73.6% Au, average).

Silver recoveries from the mixed material were variable and ranged from 25.0% to 84.2% (48.6% average). Silver recoveries tended to be higher on average (52%) for the higher sulfide grade PFS composites compared to the lower sulfide grade PEA composites (40.0% average). Silver recovery was not strongly correlated with sulfide sulfur grade.

Figure 13-2 Florida Mountain Bottle-Roll Test Recoveries, Variability Composites



Recoveries obtained from the non-oxide composites generally were low and averaged 51.4% gold and 38.0% silver.

13.3.3.5 Florida Mountain Column-Leach Testing

Seven column tests were conducted in support of the 2019 PEA using composites of 2018 drill core samples. A more extensive column testing program (19 column tests total) was conducted in support of the PFS using composites of 2019 drill core samples. All columns conducted for the PEA were run at an 80% -12.7mm (-0.5-inch) feed size. Columns conducted for the PFS study were run at a -50mm (-2-inch) feed size (10 tests), and at an 80% -12.7mm feed size (eight tests). A single test was also run on a mixed master composite of 2019 drill core samples, after single-pass HPGR crushing to a finer (77% -6.3 mm) (-0.25-inch) feed size.

All column tests were conducted without agglomeration pretreatment, but with hydrated lime (0.7 to 3.4 kg/t) blended with the feeds before leaching for pH control. Leaching conditions included solution application at a rate of 9.8 Lph/m2 at a cyanide concentration of 1.0 g NaCN/L for leach cycles ranging from 63 to 97 days for the PEA testing and from 81 to 240 days for the PFS testing. Tests were conducted to determine gold and silver recovery, recovery rate and reagent consumptions under simulated heap-leaching conditions. A summary of column-leach test results is presented in Table 13.4.

The Florida Mountain material was variable in response to simulated heap-leaching treatment (column leaching). The oxide composites generally gave very high gold recoveries (89.6% average at -12.7mm). Gold recoveries from the mixed composites were variable, with the low sulfide (<0.1% S) mixed composites giving very high gold recoveries (similar to the oxides) and the elevated sulfide (0.1% - 0.5% S) mixed composites giving gold recoveries that ranged from about 60% to 80%. It was also noted that the low sulfide mixed composites in the PEA testing tended to be from shallower material and were all "Tpr" lithology composites. Most of the elevated sulfide mixed composites of the PFS testing tended to be from deeper material and were mostly "Tql" lithology composites. As expected, the single non-oxide composite that was column tested gave a relatively low gold recovery (30.0%).

Column test silver recoveries from the -12.7 mm (-0.5-inch) feeds were significantly lower and not as variable. Silver recoveries from the oxide composites ranged from 41.3% to 52.9% and averaged 47.6%. Silver recoveries from the mixed composites ranged from 26.3% to 54.5% and averaged 41.8%. Silver recovery from the non-oxide column test was only 10.0%.

Select composites (eight total) were tested at both a -50 mm (-2-inch) feed size and at the 80% -12.7mm feed size to optimize heap-leach feed size. In the case of the oxide composites, the -50mm feed size tests were stopped before leaching was completed, after about 70 to 80 days of leaching. The corresponding -12.7mm tests were run to completion. Gold recovery rate tended to be slower at the coarser feed size and comparative rate data indicate that, allowed sufficient leaching time, gold recovery achieved at both feed sizes would be very similar. In all three cases (oxides) it was not clear that the approximately 10% difference in silver recovery between the two feed sizes would narrow significantly with more leaching time. This indicates potential for leaching of the Florida Mountain oxide material at a coarser feed size while maintaining gold recovery levels achieved at -12.7mm, but likely with moderately lower silver recoveries.

In the case of the mixed material composites tested at both feed sizes, the results were less consistent. In some cases (e.g., composite 4471-052), a large decrease in gold and silver recovery would be expected from coarsening the crush size from -12.7mm to -50mm. In other cases, (e.g., composite 4471-053), a similar ultimate gold recovery would be expected at the two feed sizes, with silver recovery expected to be substantially lower. These results indicate that the Florida Mountain mixed material is moderately to strongly sensitive to feed size for heap-leach gold and silver recoveries and requires three stages of crushing for optimal heap-leach recoveries.

A single master composite (4471-069) was prepared from the higher-grade mixed column test composites (4471-053, 4471-055 and 4471-057) on a footage weighted basis. This composite was used for a column test at a finer feed size (77% -6.3mm) (-0.25-inch) after crushing by single-pass through pilot HPGR. The HPGR sample preparation was conducted at Kappes Cassidy and Associates ("KCA"), in Sparks, Nevada (KCA, 2021). Column test results for the HPGR product showed a gold recovery (73.3%) which was approximately the same as would be expected at the -12.7mm (-0.5-inch) feed size, based on column test results from the samples comprising this feed. Silver recovery obtained from the HPGR product (63.6%) was about 12% higher than expected at a -12.7mm feed size. Replicate testing would be required to confirm the indicated improvement in silver recovery, in part because of the low-grade nature of the composite (11 g Ag/t).

Table 13-4 Column-Leach and Bottle-Roll Test Results, Florida Mountain PEA and PFS Testing

Composite	Area	Oxidation Class	Lithology Code	Sulfide Grade, % S	Test Type	Feed Size	Time, Days	Au Rec. %	Head Grade gAu/mt	Ag Rec. %	Head Grade gAu/mt	NaCN Consumed, kg/mt ore	Lime Added, kg/mt ore
PEA Testing - 2018 Drill Core Samples													
4307-132	Main - S. Central	Mixed	Tpr/Tql	0.05	CLT	12.5 mm	63	91.3	0.92	43.3	67	1.29	1.0
					BRT	1.7 mm	4	86.0	0.86	47.8	136	0.28	1.1
4307-133	Main - S. Central	Mixed	Tpr/Tql	0.02	CLT	12.5 mm	97	85.5	0.69	39.0	59	3.08	0.7
					BRT	1.7 mm	4	80.9	0.47	37.5	16	0.11	0.8
4307-135	Main - S. Central	Non-Oxide	Tql/Tpr	0.80	CLT	12.5 mm	64	30.0	0.60	10.0	10	1.22	1.8
					BRT	1.7 mm	4	35.9	0.64	20.0	10	0.15	2.0
4307-136	Main - S. Central	Oxide	Tpr	0.05	CLT	12.5 mm	63	87.2	0.39	41.3	75	1.17	1.1
					BRT*	1.7 mm	4	85.5	0.35	59.2	63	0.22	1.2
4307-137	Main - S. Central	Blend (Mix/Non-ox)	Tpr/Kgd	0.23	CLT	12.5 mm	97	65.7	1.02	30.0	170	2.03	0.5
					BRT*	1.7 mm	4	75.4	0.53	19.1	121	0.17	0.6
4307-138	Main - S. Central	Blend (Mix/Ox)	Tpr	0.04	CLT	12.5 mm	65	94.7	0.75	37.5	16	1.16	1.0
					BRT*	1.7 mm	4	88.6	0.77	28.9	18	0.22	1.1
4307-139	Main - S. Central	Blend (Mix/Ox)	Tpr/Tutbx	0.05	CLT	12.5 mm	65	90.2	0.61	26.3	19	1.18	1.0
					BRT*	1.7 mm	4	82.6	0.78	44.8	28	0.16	1.4
PFS Testing - 2019 Drill Core Samples													
4471-048	Main - N./Central	Ox	Tpr/Tql	<0.01	CLT	-50mm	81	82.4	0.51	30.8	13	1.07	1.6
					CLT	12.5mm	207	94.3	0.53	52.9	17	2.25	1.6
					BRT	1.7mm	4	93.3	0.45	54.5	14	0.12	1.8
4471-049	Main - South	Ox	Tpr/Tql	<0.01	CLT	-50mm	81	78.8	0.33	33.3	9	1.08	1.7
					CLT	12.5mm	207	89.7	0.39	50.0	10	2.93	1.7
					BRT	1.7mm	4	91.7	0.24	53.8	10	0.06	1.9
4471-050	Tip - Top	Blend (ox/mix)	Tpr/Tql	0.02	CLT	-50mm	70	80.0	0.25	23.1	13	0.80	1.4
					CLT	12.5mm	175	87.0	0.23	46.2	13	1.59	1.4
					BRT	1.7mm	4	90.0	0.20	40.6	14	0.07	1.6
4471-051	Main - North	Mixed	Tpr/Tql	0.06	CLT	-50mm	240	59.3	0.27	20.0	10	1.90	3.1
					BRT	1.7mm	4	55.0	1.11	40.0	9	0.26	3.4
4471-052	Main - N./Central	Mixed	Tql (some Tpr)	0.36	CLT	-50mm	179	46.6	1.03	24.5	49	1.47	1.5
					CLT	12.5mm	179	73.6	1.10	50.9	55	2.86	1.5
					BRT	1.7mm	4	69.8	1.26	61.6	64	0.07	2.4
4471-053	Main - N./Central	Blend (mix/unox)	Tql/Tpr	0.39	CLT	-50mm	215	66.7	0.36	42.1	19	1.48	1.5
					CLT	12.5mm	147	67.4	0.43	54.5	22	1.71	1.5
					BRT	1.7mm	4	77.3	0.22	69.6	30	0.20	1.9
4471-054	Main - North	Mixed	Tql (some Tpr/Tuff/Tutbx)	0.25	CLT	-50mm	148	43.2	0.44	40.0	5	0.96	2.3
					BRT	1.7mm	4	72.4	0.29	61.3	8	0.13	2.5
4471-055	Main - South	Mixed	Tql (some Ttb/Tpr)	0.51	CLT	-50mm	148	48.8	0.43	38.5	13	1.33	1.7
					CLT	12.5mm	186	71.2	0.73	54.5	11	2.98	1.7
					BRT	1.7mm	4	78.0	0.41	64.6	10	0.27	1.9
4471-056	Main - South	Mixed	Tql/Tpr	0.14	CLT	-50mm	203	65.4	0.26	25.0	16	1.91	2.0
					CLT	12.5mm	156	62.5	0.24	45.5	11	1.87	2.0
					BRT	1.7mm	4	73.9	0.23	58.1	9	0.1	2.2
4471-057	Tip - Top	Blend (mix/ox)	Tpr	0.17	CLT	-50mm	148	69.2	0.26	33.3	9	1.27	1.6
					CLT	12.5mm	156	76.0	0.25	36.4	11	1.71	1.6
					BRT	1.7mm	4	75.0	0.24	53.2	9	0.16	1.8
4471-065		Mixed	Blend	0.40	CLT	HPGR	126	73.3	0.45	63.6	11	1.36	1.6

Note: Blend denotes more than one material type (either oxidation class or lithology) was included in the composite.
CLT denotes column leach test. BRT denotes bottle roll test. HPGR denotes that test was conducted on product generated using high pressure grinding rolls.
-50 mm feeds were crushed to just passing 50mm. 12.5mm feeds were crushed to 80%-12.5mm and 1.7mm feeds were crushed to 80%-1.7mm.

13.3.3.6 Florida Mountain Load/Permeability Testing

Select column leached residues (-12.7mm feed size) generated during the PEA and PFS testing were submitted to Geo-Logic Associates, in Sparks, Nevada, for load/permeability tests (test procedure USBR 5600-89). The tests were conducted to evaluate permeability of the leached material under expected commercial heap stack heights. Results showed generally high permeabilities over a range of simulated heap stack heights of up to 137 meters (449.5 feet) (McPartland, 2022). These results demonstrate that the Florida Mountain oxide and mixed materials generally will not require agglomeration pretreatment for successful heap leaching at heap stack heights of up to about 150 meters (492 feet).

13.3.4 DeLamar and Florida Mountain Mill Testing

The PFS mill testing was conducted at McClelland in multiple stages, primarily on non-oxide material types. Direct agitated cyanide leaching was evaluated at feed sizes ranging from 80% -75 microns (200 mesh) to 80% -10 microns to evaluate gold and silver liberation characteristics and the potential for very fine or ultra-fine regrinding of flotation concentrate followed by agitated cyanidation.

The report by Wickens (2020) summarized all testing to December 2020 on DeLamar and Florida Mountain samples at McClelland. The PEA metallurgical testing was described as Test Series 1 through 13 in Wickens (2020). PFS testing related to flotation and flotation concentrate processing conducted through December 2020 was described as Test Series 13, 14 and 18 through 21 in Wickens (2020). Those test series designations are mentioned here where appropriate for ease of reference. Mill testing conducted at McClelland in 2021 as part of the PFS metallurgical testing program was described in a separate report (McPartland, 2021d) and in the summary report (McPartland, 2022).

13.3.4.1 DeLamar Mill Testing

DeLamar PEA Mill Testing

Testing conducted at McClelland during the 2019 PEA study (Gustin et. al., 2019) included evaluation of grind-leach, gravity concentration and flotation processing of the DeLamar non-oxide materials. Testing to optimize flotation processing was also conducted.

Thirty non-oxide composites from DeLamar, Glen Silver, Sommercamp and Sullivan Gulch, along with three mixed, one oxide and two blended (mixed/non-ox) 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. Test results showed that the oxide and mixed material types were amenable to milling/cyanidation treatment, at an 80% - 75µm feed size. Gold recoveries ranged from 61.8% to 88.9%. The non-oxide material generally was not amenable to milling/cyanidation treatment at the same feed size.

A series of scoping-level gravity concentration tests, with bulk sulfide flotation on the gravity tailing, was conducted on nine non-oxide drill core composites from Sullivan Gulch and Glen Silver. 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 tailing 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 at 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.

Two master composites were prepared from each of the non-oxide 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. The 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. 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%. It was noted that gold and silver recoveries in excess of 90% are expected to be obtainable at a concentrate mass pull of about 10% to 13%.

DeLamar PFS Mill Testing

Multiple flotation tests were conducted on a total of 26 drill hole composites, as part of the McClelland PFS metallurgical study. These tests were conducted to further evaluate variability in response to flotation treatment and to generate flotation concentrate for cyanidation testing. Summary results for flotation tests conducted during the PFS testing are presented in Figure 13.3 and Figure 13.4.

Figure 13-3 Glen Silver Flotation Recoveries, PFS Composites

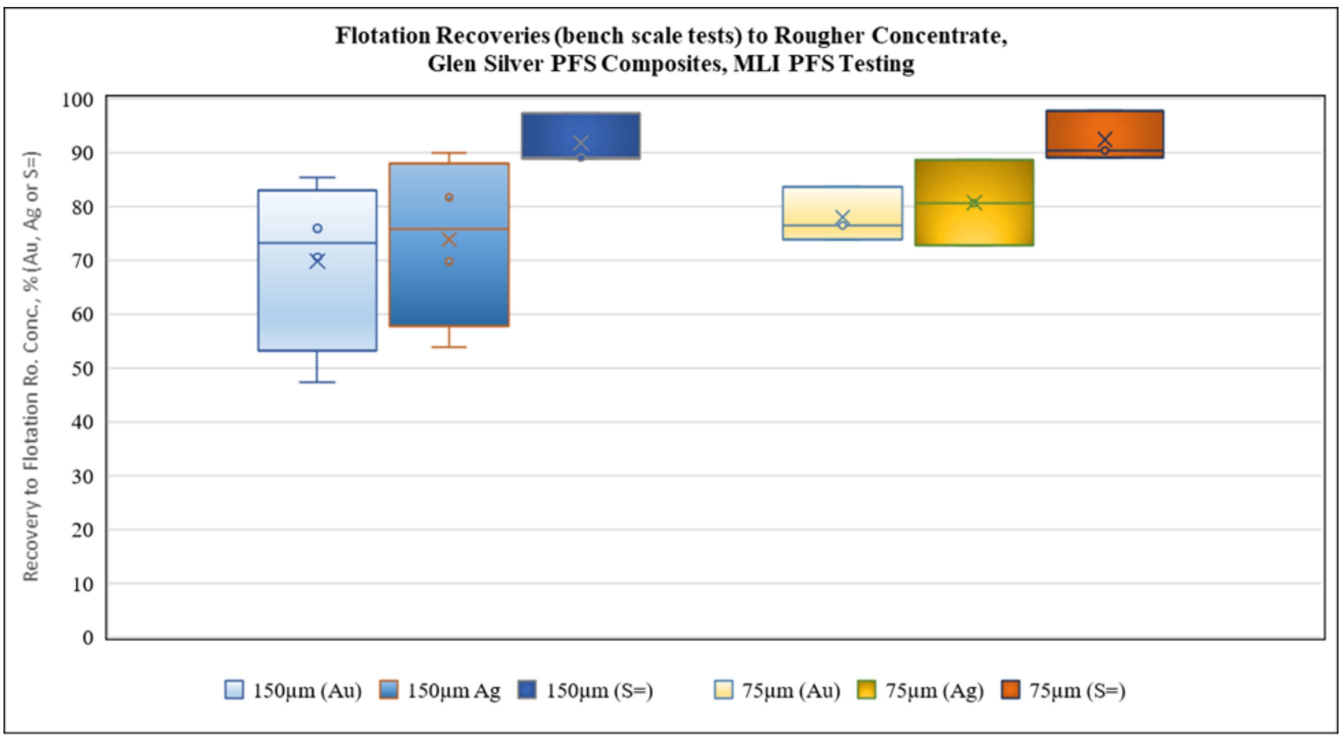
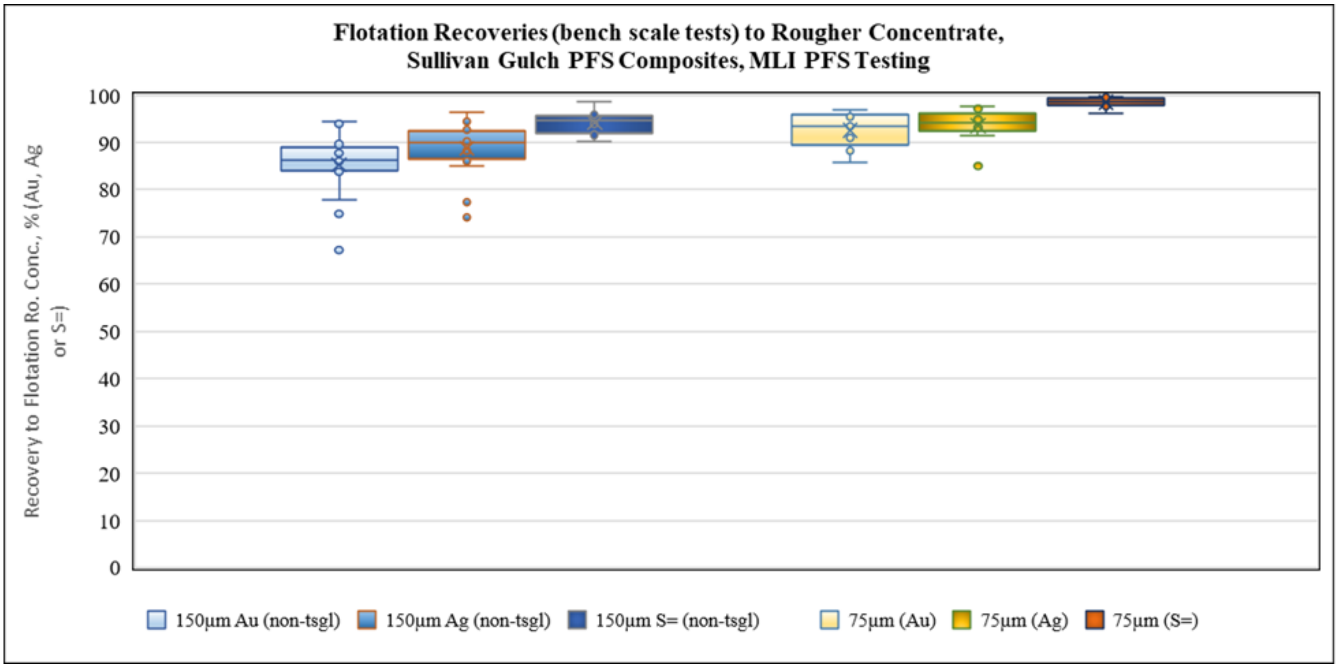


Figure 13-4 Sullivan Gulch Flotation Recoveries, PFS Composites



Flotation testing generally showed that the Glen Silver composites (four), Sommercamp composite (one) and Sullivan Gulch composites (21) responded well to conventional flotation treatment, at either an 80% -150 microns (100 mesh) or 80% -75 microns (200 mesh) feed size. When the same composite was evaluated at both feed sizes, only done for three Glen Silver composites, recoveries to the rougher concentrate were essentially the same at the two sizes. Rougher flotation mass pulls were variable and averaged 13% for the -150 micron feeds and 17% for the -75 micron feeds.

Flotation gold recoveries that reported to the concentrates produced from the Glen Silver composites generally were equivalent to between 70.5% and 85.3%. Flotation gold recoveries averaged 77.2% at the 150 microns feed size and 78.0% at the 75 microns feed size, for the three composites tested at both feed sizes. Silver recoveries obtained from the Glen Silver composites generally ranged from 69.7% to 89.9%. Flotation silver recoveries averaged 80.4% at the 150 microns feed size and 80.7% at the 75 microns feed size, for the three composites tested at both sizes.

A single Sommercamp composite was evaluated at an 80% -150 microns feed size. The flotation test with a 10.4% mass pull gave rougher concentrate gold and silver recoveries of about 90%. The second test had a lower mass pull and correspondingly lower recoveries.

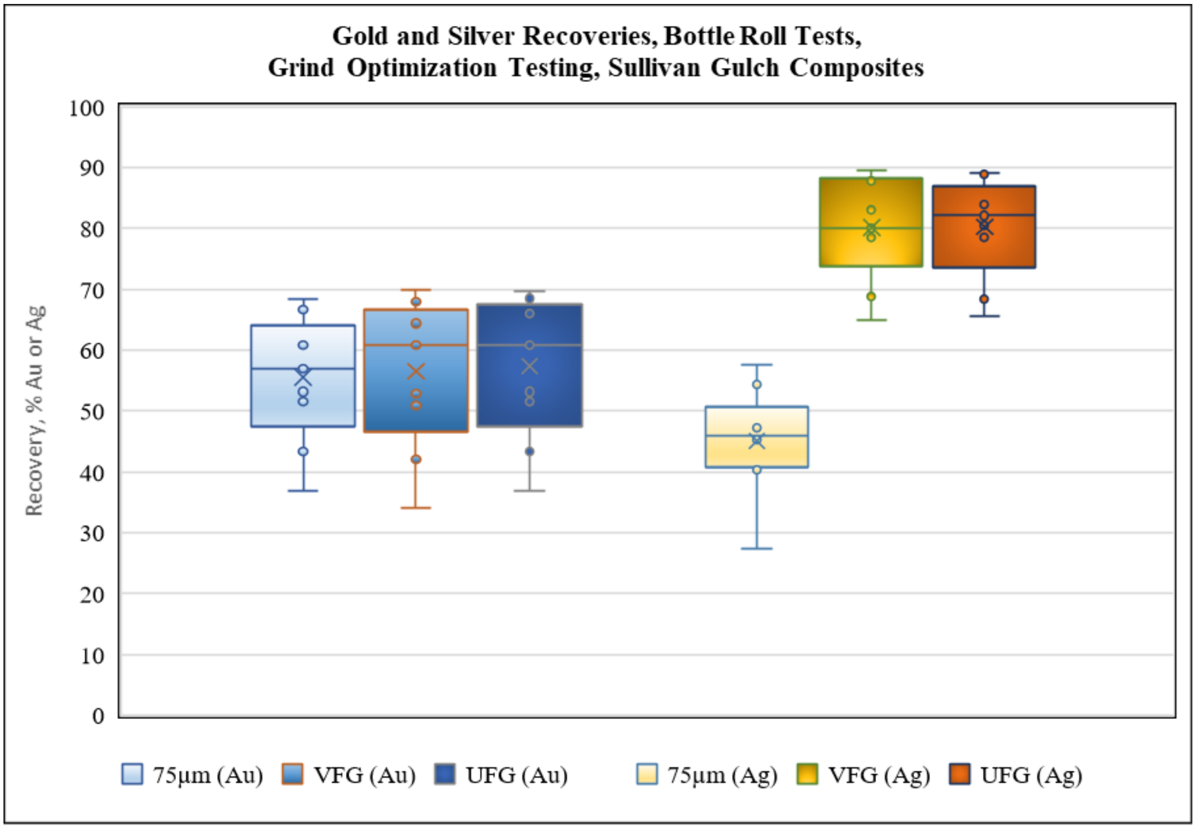
A total of 15 Sullivan Gulch composites were used for flotation testing at an 80% -150 microns feed size and nine were used for flotation testing at an 80% -75 microns (200 mesh) feed size. None of the composites were tested at both feed sizes. Samples tested at the -150 microns (100 mesh) feed size included two Sullivan Gulch latite ("Tsgl") lithology composites and one composite that contained 40% Tsgl lithology material. The Tsgl lithology composites gave significantly poorer flotation response than any of the other samples tested. Mass pulls for this material tended to be high and recoveries tended to be low and variable where more than one test was conducted. It was noted during these tests that there was a significant carry-over of clay-like gangue material into the rougher concentrates. Analysis of drill hole logging by Integra staff indicated that the Tsgl lithology represents a relatively small portion, 3% by drill hole length, of the Sullivan Gulch non-oxide material above cut-off grade.

Nine Sullivan Gulch master composites were prepared by combining earlier composites, on a footage weighted basis, according to drill holes, for rougher flotation testing. A rougher flotation test was conducted on each composite at an 80% -75 microns (200 mesh) feed size, using procedures optimized during earlier testing.

Gold recovery ranged from 91.0% to 96.8% for seven of the nine composites. Gold recovery for the other composites were 85.8% and 88.2%. Silver recoveries ranged from 96.2% to 99.6% for all but one composite at 85.0%. Mass pulls to rougher concentrate were relatively high and ranged from 9.2% to 27.3% (17.3% average). Sulfide sulfur recoveries were very high (98.5% average) and indicate that the gold and silver recoveries are probably near the maximum achievable by flotation.

A series of cyanidation tests were conducted on "whole ore" feed from 10 of the master composites used for the flotation testing described above, along with another Sullivan Gulch composite (4307-201), to investigate the effects of very fine grinding ("VFG") and ultra-fine grinding ("UFG") on gold and silver recoveries by cyanidation. The composites tested were from Glen Silver (one) and Sullivan Gulch (10). The test protocol consisted of a grind-leach cyanidation test on a 1.0 kg sample at an 80% -75 microns (200 mesh) feed size, followed by regrinding and leaching sub-samples split from the grind-leach tailing. Each grind-leach tail sample was used for two regrind-releach tests; one at an approximately 80% -20 microns regrind size (designated very fine grinding) and one at an approximately 80% -11 microns regrind size (designated ultra-fine grinding). Particle size distributions ("PSDs") were determined by laser size analysis, on select leached residues. In the case of the tests where a PSD was not determined, the average of results (P_{80}) from the measured PSDs was assumed (~ 20 microns for VFG and 11 microns for UFG. Results from the test series are shown in Figure 13.5.

Figure 13-5 Grind-Cyanide Leach Testing Gold and Silver Recoveries, Sullivan Gulch Composites



For the Glen Silver composite, very fine grinding to approximately 80% -20 microns had little or no effect on gold recovery, compared to leaching at the 75 microns feed size. Ultra-fine grinding to approximately 80%-10 microns appeared to improve gold recovery by about 9%, compared to that obtained at 75 microns or 20 microns. Glen Silver composite silver recoveries for VFG and UFG were similar and were significantly higher than the 75 microns grind size.

Silver recovery from the Glen Silver composite improved from 44.3% at the 75 microns feed size to 66.7% for the VFG test. Finer grinding (~ 11 microns) did not further improve silver recovery.

For the Sullivan Gulch composites, gold recoveries for the VFG and UFG tests were similar, and VFG or UFG resulted in a small improvement in gold recovery. Average gold recovery from the Sullivan Gulch composites for the 75 microns (54.0%), VFG (56.5%) and UFG (57.4%) tests were essentially the same. For all three feed sizes, gold recovery tended to increase with increasing CN/FA ratio.

Silver recoveries for Sullivan Gulch composites were significantly improved by very fine grinding to approximately 20 microns in size. At the 75 microns feed size, seven of the 10 samples had silver recoveries between 40% and 50%. The remaining three composites had silver recoveries of 54.3%, 57.6%, and 27.4%. After very fine grinding eight of the 10 composites gave silver recoveries of >78% with an average of 83.7%. The remaining two composites gave silver recoveries of 65.0% and 68.8%. Grinding from approximately 20 microns to 11 microns did not significantly further improve silver recoveries. Average silver recovery for the nine Sullivan Gulch composites was 80% for both the VFG and UFG tests.

A total of 12 DeLamar non-oxide drill core composites were prepared for flotation testing and flotation regrind/cyanidation testing to evaluate response to processing optimized for the DeLamar non-oxide material. Ten of the twelve composites were comprised of material from the Sullivan Gulch area. The two remaining composites represented material from the Glen Silver area (4307-193/194) and from the Sommercamp area (4522-085).

Two flotation tests were conducted on each composite, using conditions optimized during earlier testing, to confirm response to flotation treatment and to generate concentrate for concentrate regrind/-cyanidation tests. Feed size for the flotation tests was 80% -150 microns (100 mesh). Only one test was possible on composite 4307-193/194, because of sample limitations. Agitated cyanidation tests were conducted on flotation concentrate generated from each of the composites after fine regrinding. Regrind sizes ranged from 80% -7.0 microns to 80% -39 microns and averaged 80% -17.7 microns. Those results are shown in Table 13.5.

Table 13-5 Flotation Concentrate Regrind/Cyanidation Tests, DeLamar Non-Oxide Composites

		Regrind	Cyanidation Recovery		Head Grade		Reagents, kg/t	
Composite	Area	P_{80} (µm)	% Au	% Ag	g Au/t	g Ag/t	NaCN	Lime
4522-074	Sullivan Gulch	22.8	11.1	87.3	1.89	79	3.93	4.1
4522-075	Sullivan Gulch	39.0	14.4	69.4	0.90	36	2.94	3.1
4522-076	Sullivan Gulch	15.7	27.8	86.6	3.56	224	7.07	4.4
4522-077	Sullivan Gulch	9.0	13.4	86.3	13.98	1347	8.36	6.7
4522-078	Sullivan Gulch	14.8	13.8	83.9	2.10	161	3.63	5.6
4522-079	Sullivan Gulch	16.1	32.5	87.6	2.37	129	11.35	4.2
4522-080	Sullivan Gulch	18.2	29.5	84.2	4.57	715	5.81	3.1
4522-082	Sullivan Gulch	21.0	38.0	84.9	2.21	770	3.97	4.4
4522-083	Sullivan Gulch	15.0	27.1	67.9	2.91	56	4.81	4.8
4522-084	Sullivan Gulch	7.0	31.2	88.0	5.73	191	8.4	8.4
4522-085	Sommercamp	11.2	44.5	94.2	2.54	813	6.92	6.3
4307-193/194	Glen Silver	22.8	61.5	53.3	1.69	30	5.37	7.0

Note: Recoveries and reagent consumptions are presented on a flotation concentrate basis.

The concentrates were not particularly amenable to cyanidation for recovery of contained gold. Gold recovery obtained from the Sullivan Gulch concentrates ranged from 11.1% to 38.0% and averaged 23.9%. These recoveries were equivalent to between 4.5% and 30.7% and averaged 18.5% of gold contained in the flotation feed. This compares to an average CN/FA for these composites of 8.8% gold.

Gold recoveries obtained from the Sommercamp and Glen Silver concentrates were somewhat higher at 44.5% and 61.6%, respectively, and were equivalent to 33.2% and 29.1%, respectively, of gold contained in the flotation feed. Respective CN/FA ratios for these composites were 19.2% and 23.9%. Concentrate cyanidation gold recoveries were not sensitive to regrind size, within the range evaluated.

Silver recoveries obtained from the Sullivan Gulch concentrates were consistently higher and ranged from 67.9% to 88.0% with an average of 82.6%. These recoveries were equivalent to between 46.6% and

67.3% (56.2% average) of silver contained in the Tsgl composites and to between 62.0% and 77.8% averaging 72.5% of silver contained in the other Sullivan Gulch composites. Silver recovery from the Sommercamp concentrate was quite high at 94.2% and was equivalent to 84.8% of silver contained in the flotation feed. Silver recovery from the Glen Silver composite was lower at 53.3% and was equivalent to 28.7% of silver contained in the flotation feed.

Reagent consumptions for the concentrate leach tests generally were moderate. Cyanide consumption for the Sullivan Gulch concentrates ranged from 2.94 to 22.53 kg NaCN/t concentrate, which was equivalent to between 0.33 and 2.30 kg NaCN/t of flotation ("whole ore") feed. Lime required for the Sullivan Gulch concentrates was equivalent to between 3.1 and 25.0 kg/t concentrate, which was equivalent to between 0.3 and 2.6 kg/t flotation feed. Cyanide leaching conditions were not optimized, and it is expected that reagent consumptions can be decreased through optimization testing.

DeLamar PFS Flotation Concentrate Albion Process and Roaster Testing

Glen Silver composite 4307-202 and Sullivan Gulch composite 4307-212 were used to create flotation concentrate for third-party Albion and roasting testing. Five bulk flotation tests, 8.0 kg (17.6 pounds) each, were run to generate sufficient concentrate for these tests. Flotation test mass pulls were relatively high (14.9% and 19.2%). Gold, silver, and sulfide sulfur recoveries were about as expected based on earlier testing on the same composite (Comp. 4307-202) and on earlier testing on the constituent composites comprising the master composite tested (Comp. 4307-212). Silver recovery to the bulk rougher concentrate produced from Glen Silver composite 4307-202 (63.8%) was lower than obtained during an earlier 1.0 kg test (81.7%). Baseline cyanidation testing (discussed below) confirmed that the concentrates were refractory to cyanidation treatment, particularly with respect to gold recovery.

The two flotation concentrates were used for scoping level evaluation of the Albion process for recovery of gold and silver from the flotation concentrate. The Albion testing was performed by SGS Minerals in Lakefield, Ontario (Geldart, 2020). The proprietary Albion process employs ultra-fine grinding + atmospheric oxidation followed by agitated cyanidation. Test results showed that the Albion process significantly improved gold recovery for both concentrates. Cyanidation gold recovery from the Glen Silver rougher concentrate increased from 34.8% (baseline CIL) to 91.5% after Albion pretreatment. The Sullivan Gulch rougher concentrate gold recovery increased from 62.5% to 93.0%. Silver recovery for the Glen Silver rougher concentrate increased from 66.6% to 95.0%. Silver recovery for Sullivan Gulch rougher concentrate was very high without pretreatment (93.4%) and essentially the same after pretreatment (94.2%). Sulfide oxidation levels achieved for the Glen Silver and Sullivan Gulch concentrates were reported as 82.2% and 88.6%, respectively. These results demonstrate that sulfide mineral oxidation using the Albion process is effective in liberating gold contained in the DeLamar flotation concentrates, and that the pretreated concentrates give high gold and silver recoveries by cyanide leaching.

Comparative roaster tests were performed on splits from the same flotation rougher concentrates by Jerritt Canyon Gold ("JCG") at the Jerritt Canyon mine site laboratory in Elko County, Nevada (Bond, 2020). Baseline CIL tests were conducted on the concentrate after regrinding to 80% -68µm to 69µm. Baseline CIL gold recoveries were reported as 20.6% (4307-FC-202A) and 11.7% (4307-212A). Because of the high sulfide sulfur content of the concentrates (12.6% - 13.4%), the flotation concentrates had to be blended with Jerritt Canyon ore to meet the roaster's designed operating conditions. Calculated CIL gold recoveries from the DeLamar concentrates after roasting were high (>90%) and indicated that gold contained in the concentrates was liberated for cyanide leaching during roasting. These results also demonstrate that oxidation of contained sulfide minerals is effective in liberating gold contained in the DeLamar flotation concentrates and rendering the gold amenable to recovery by cyanide leaching.

DeLamar Post-PFS Mill Testing

A detailed flotation testing program was conducted in 2022 following the publication of the PFS technical report to optimize flotation processing for the DeLamar non-oxide material and to evaluate ore variability with respect to flotation processing. That work was discussed in a 2023 report (McPartland, 2023a). A total of 48 variability composites and three master composites were evaluated. The composites represented material from the Sullivan Gulch (SG), Ohio Sullivan Gulch (OH/SG), Milestone (MS), DeLamar North (DLMN), South Wahl (SW) and Sommercamp (SC) ore zones of the DeLamar project.

Initially, the 48 variability composites were prepared and subjected to detailed head assay. These composites included approximately 790 lineal meters of drill core from 12 drill holes. Composites were prepared based on ore zone, location, lithology, and where possible alteration. Splits from the variability composites were used to prepare the four master composites, which were used for flotation optimization testing. These composites were designated 4750-052 (SG low cyanide solubility), 4750-053 (SG high cyanide solubility), 4750-057 (SG overall) and 4750-054 (other areas, i.e., non-SG and OH/SG material). The SG master composites included the OH/SG materials.

Head assays showed that the variability composites contained between 0.08 and 1.20 g Au/mt (0.41 g Au/mt avg.) and between 3 and 249 g Ag/mt (44 g Ag/mt avg.). Cyanide soluble gold and silver content generally were low and averaged 24% and 35%, respectively, of the assayed head grades. Sulfide sulfur content was variable and averaged 1.91%.

Textural mineralogical analysis was conducted on the master composites and on select variability composites. Results showed that the master composites contained elevated levels of illite, with small amounts of other clays. Total clay content by mass was estimated to be 10.3% (SG low CN Sol), 13.2% (SG high CN Sol) and 21.3% (other areas). Slurry rheology testing was conducted on each of the same three master composites and indicated slurry viscosity characteristics were acceptable for flotation processing at solids densities of approximately 33% or lower.

A total of 55 rougher flotation tests were conducted on the four master composites to optimize primarily reagent suites (type and dose). Almost all the tests were conducted at an 80%-150µm feed size, which was optimized during earlier flotation testing programs. Results showed that for the SG material, it was possible to recover >80% of the gold, and approximately 90% of the silver and sulfide sulfur to a concentrate of about 15% mass using several reagent schemes. This testing included the evaluation of multiple collector combinations, clay modifiers, dispersants and frothers. Testing indicated that the ore was sensitive to flotation pH and adding caustic equivalent to approximately 1.5 kg/mt NaOH enhanced flotation response when using xanthate and dithiophosphate collectors. Activation with copper sulfate generally was necessary to maximize recoveries. Similar gold and silver recoveries were obtained using a collector combination of Solvay’s A412 and A5100 collectors with copper sulfate activation, without adding caustic. That reagent suite was selected for ore variability testing.

Similar flotation results were obtained from the other areas master composite (4750-054), but with somewhat lower recoveries. Rougher flotation recoveries of >70% gold and silver were obtained with sulfide sulfur recoveries of >90% to rougher concentrates with a mass pull of approximately 15%. It should be noted here that approximately 33% of the mass and 54% of the gold contained in this composite came from MS material. As discussed below, MS material responded poorly to flotation treatment under the conditions tested, which contributed to the relatively lower recoveries obtained from the other areas master composite. The same reagent scheme as selected for the SG composites was selected for the other areas variability testing.

A single rougher flotation test was conducted on each of the 48 variability composites to assess flotation response and variability. Tests were conducted using the optimized reagent suite at an 80%-150µm feed size. Recoveries generally were lower than obtained during optimization testing and indicate that optimal flotation conditions will probably be variable for the various material types. In some cases (presumably high clay materials), slurries appeared to be relatively viscous and more difficult to process by flotation. Average flotation gold recoveries (number of samples tested in parentheses) obtained for the SG (23), OH/SG (12), MS (10), NDLM (1), SW (1) and SC (1) composites were 69%, 64%, 46%, 84%, 51% and 84%, respectively. Corresponding average silver recoveries were 81%, 78%, 59%, 84%, 56% and 87%, respectively.

Response of the Milestone composites to flotation appeared to be anomalous compared to the other zones, with low precious metal recoveries (see above) and sulfide sulfur recoveries (63% avg.). The current work represents the first known flotation testing on this material. Further testing, including mineralogy and feed size optimization, will be required for the Milestone material to optimize flotation response.

No clear trends relating flotation response to ore lithology (rock unit), or alteration were observed. Spatial variations (distinguished by drill hole) in recovery were observed within each of the SG, OH/SG and MS zones.

Further flotation testing should focus on development of a more robust reagent scheme, capable of maintaining recoveries for the higher clay material. Use of a pH modifier/dispersant such as caustic soda may be required for the higher clay materials. When processing high clay ores, ore blending with more competent material may be required.

13.3.4.2 Florida Mountain Mill Testing

Florida Mountain PEA Mill Testing

Testing in support of the 2019 PEA was conducted mainly on 10 non-oxide drill core composites and one overall “master” non-oxide drill core composite. That testing included “whole ore” grind/agitated cyanidation (bottle-roll) tests, gravity concentration followed by cyanide leaching of the gravity tails, and gravity concentration followed by flotation and regrind/cyanidation of the flotation concentrate. Ten of the 2018-2019 Florida Mountain non-oxide composites, along with one of the mixed composites were used for bottle-roll leach tests at an 80% -75 microns (200 mesh) feed size to evaluate amenability to grind-leach (cyanidation) treatment. These tests were conducted using a 72-hour leach cycle, at a solids density of 40% and a cyanide concentration of 1.0 g NaCN/L.

All of the non-oxide Florida Mountain composites tested were amenable to grind-leach treatment, at an 80% -75 microns 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.

A single gravity-concentration test was conducted on a Florida Mountain non-oxide master composite (4307-160) to evaluate response to gravity concentration and to generate gravity tailing 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 tailing, generated as described in the preceding paragraph, at five tailing regrind sizes ranging from 80% -150 microns (100 mesh) to 80% -45 microns (325 mesh). The gravity tailing was amenable to agitated cyanidation treatment at the regrind sizes tested. Combined gold recovery (gravity concentration + tailing 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.

Bulk sulfide flotation tests were also conducted on separate splits of the same gravity tailing, at regrind sizes ranging from 80% -212 microns (65 mesh) (no regrind) to 80% -75 microns (200 mesh), to evaluate the potential for upgrading the gravity tailing by flotation. The gravity tailing 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 106 microns, 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”.

A larger, 8-kilogram flotation test was conducted on the same gravity tailing (non-oxide master composite 4307-160), at an 80% -212 microns 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 tailing. The flotation rougher concentrate produced was used as feed for an intensive cyanidation test.

The flotation rougher concentrate produced from the Florida Mountain non-oxide master composite 4307-160 was readily amenable to agitated cyanidation treatment, at a 95% -37 microns 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 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 compared favorably to those obtained by gravity concentration with cyanidation of the gravity tailing. Gold and silver recoveries obtained from the master composite by gravity concentration with cyanidation of the gravity tailing did not exceed 82.2% and 71.3%, respectively, at gravity tailing regrind sizes as fine as 80% -45 microns (325 mesh). 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 effective for processing of the Florida Mountain non-oxide material.

Florida Mountain PFS Mill Testing

Detailed mill testing was conducted on a total of 10 non-oxide composites from the Florida Mountain deposit, to optimize processing conditions for recovery of gold and silver by flotation and agitated cyanidation of the flotation concentrate, and to evaluate ore variability. Initially seven composites were prepared from five drill holes, including two from the Tip-Top area and three from the North to Central portion of the Main area and one from the South to Central portion of the Main area. Scoping agitated cyanidation tests were conducted on each composite at 80% -1.7mm feed size and at an 80% -75 microns (200 mesh) feed size. Each was leached for 96 hours, using a solids density of 40% and a cyanide concentration of 1.0 g NaCN/L.

Bottle roll test results showed that the composites were amenable to agitated cyanidation treatment. Gold recoveries (44.4% - 70.9%) and silver recoveries (30.2% - 50.0%) obtained at the 1.7mm (10 mesh) feed size indicate some potential for heap leaching of the non-oxide material but expected recoveries would be significantly lower than obtained from oxide or mixed materials.

Results from the grind-leach cyanidation tests confirmed that the non-oxide material was not refractory to cyanidation treatment. Gold recoveries obtained in 4 days of leaching ranged from 70.0% to 89.5% (81.9% average). Silver recoveries ranged from 43.8% to 63.0% (55.4% average).

A scoping gravity concentration test was conducted on a 12 kg (24.5 pound) split from each of the seven composites, at 80% -212 microns (65-mesh), to evaluate response to gravity concentration and to generate gravity tailing for flotation testing.

The composites responded moderately well to gravity concentration. Gold values reporting to a gravity cleaner concentrate with an average mass of 0.09% of the feed and an average grade of 146 g Au/t, represented gold recoveries of between 19.4% and 65.3% (38.0% average). Gravity gold recovery was not correlated to feed grade. These results indicate moderate potential for gold recovery by gravity concentration. Silver recoveries were much lower and did not exceed 11% to the cleaner concentrate.

Bulk sulfide flotation tests were conducted on each of the seven non-oxide composites, at an 80% -212 microns (65 mesh) feeds size, both with and without head-end gravity concentration treatment. Based on results from those tests, two master composites (4471-067 and 068) were prepared and used along with one of the original composites (4471-061) for more detailed testing. That testing included a series of grind optimization flotation tests on gravity tailing and a comparative whole ore flotation test at the indicated optimum grind size (80% -150 microns) (100 mesh). Based on results from these tests an overall master composite (4471-069) was prepared for bulk flotation, to generate concentrate for cyanidation testing.

The flotation variability testing showed that the non-oxide composites generally responded well to direct flotation treatment, at an 80%-212 microns feed size. Gold values reporting to the flotation rougher concentrates (5.3% mass pull average) generally ranged from 72% to 95% and averaged 78.0% of the total gold contained in the feed. Direct flotation silver recoveries obtained at the 212 microns feed size were significantly lower and ranged from 23.7% to 66.4%. Sulfide sulfur recoveries to the rougher concentrate (measured only for the tests without gravity pre-concentration) were consistently high and averaged 90%.

In summary, flotation test results indicated that a feed size of 80%-150 microns was optimum for maximizing flotation gold and silver recovery from the Florida Mountain non-oxide mineralization. Gravity pre-concentration resulted in negligible improvement of gold and silver recoveries to the concentrate at this feed size.

Agitated cyanidation tests were conducted on flotation rougher concentrate, generated from the bulk flotation tests, to optimize regrind size for cyanide leaching of the concentrate. Regrind sizes ranging from 80% -39 microns to 80% -14 microns were tested. Cyanidation test conditions and results are shown in Table 13.6

Table 13-6 Florida Mountain Flotation Concentrate Regrind/Cyanidation

(Non-Oxide Master Composite 4471-069)

72 hour leach cycle, 30% solids density, pH 11.0 - ll.5 using lime, 5.0 gNaCN/L solution										
Regrind	Au	gAu/mt Flot Conc.			Ag	g Ag/mt Flot. Conc.			Reagents	
P_{80}	Recovery,			Head	Recovery,			Head	kg/mt Flot Conc.	
µm	%	Extracted	Tail	Grade	%	Extracted	Tail	Grade	NaCN Cons.	Lime Added
39	85.6	4.09	0.69	4.78	75.0	135	45	180	5.76	2.3
35	87.8	5.32	0.74	6.06	80.1	161	40	201	4.52	2.2
29	86.8	4.67	0.71	5.38	79.8	142	36	178	4.52	2.3
20	88.2	4.58	0.61	5.19	87.2	156	23	179	5.94	1.6
14*	89.5	4.86	0.57	5.43	87.3	144	21	165	5.69	2.4
* Lead nitrate added initially at 0.05 kg/mt concentrate.										
Note: Flotation rougher concentrates were generated at an 80%-150µm feed size.										

Results showed that gold recovery increased incrementally with decreasing regrind size from 85.6% of gold contained in the concentrate (80% -39m) to 89.5% (80% -14 microns). Silver recovery was more sensitive to regrind size and increased from 75.0% (80% -30 microns) to 87.2% (80% -20 microns) of silver contained in the concentrate. Grinding from 20 microns to 14 microns did not further increase silver recovery.

Based on the average flotation recoveries (86.5% Au and 72.4% Ag) to the bulk rougher concentrates used for these cyanidation tests, the concentrate leach recoveries represented overall (flotation/ concentrate leach) recoveries between 74.0% and 77.7% gold and between 54.3% and 63.2% silver. Considering the flotation testing on the individual composites discussed in this section, higher flotation recoveries are expected from these materials in a continuous flotation circuit. Reagent consumptions were relatively low and equivalent to 0.30 kg NaCN/t and 0.1 kg/t on a “whole ore” basis, at the 20 microns regrind size.

13.4 DeLamar and Florida Mountain PFS Recovery and Reagent Estimates

The recovery and reagent consumption estimates used for the 2022 PFS study are summarized in this section of the report. These estimates have not been updated from the 2022 PFS report.

13.4.1 Heap Leach Recovery Estimates

Expected heap-leach gold recoveries for the oxide mineralization from both deposits (DeLamar and Florida Mountain) are consistently high (70% - 89%). Heap leach gold recoveries for the mixed mineralization are expected to average 72% for Florida Mountain and to range from 45% to 63% for the DeLamar deposit. Heap leach silver recoveries from the Florida Mountain oxide and mixed materials are expected to average 49% and 47%, respectively. Expected heap-leach silver recoveries from the DeLamar material are highly variable (11% to 74%), but generally low. A significant portion of the DeLamar oxide and mixed mineralization will require agglomeration pretreatment using cement, because of elevated clay content. None of the Florida Mountain heap-leach material is expected to require agglomeration. Estimated recoveries and reagent consumptions for heap leaching of the DeLamar and Florida Mountain oxide and mixed material types are shown in Table 13.7.

Table 13-7 PFS Heap-Leach Recovery and Reagent Consumption Estimates

PFS Heap Leach Recovery and Reagent Consumption Estimates						
Zone	***Oxidation***	***Au Rec %***	***Ag Rec %***	***NaCN (kg/t)***	***CaO (kg/t)***	***Cement (kg/t)***
Florida Mountain	Oxide	89	49	0.4	2.0	0
Florida Mountain	Mixed	72	47	0.6	2.4	0
DeLamar/DeLamar North	Oxide	78	11	0.3	0.4	2.8
Glen Silver	Oxide	70	18	0.4	0.5	2.8
Sullivan Gulch/Ohio	Oxide	86	20	0.5	0.7	2.8
Sommercamp	Oxide	87	15	0.3	0.3	2.8
South Wahl	Oxide	77	37	0.4	0.7	2.8
Milestone	Oxide	75	18	0.4	0.3	2.8
DeLamar/DeLamar North	Mixed	61	42	0.5	0.9	2.8
Glen Silver	Mixed	63	30	0.5	0.6	2.8
Ohio/Sullivan Gulch	Mixed	61	39	0.5	0.9	2.8
Sommercamp	Mixed	58	44	0.5	0.9	2.8
South Wahl	Mixed	50	74	0.4	1.1	2.8
Milestone	Mixed	45	18	0.4	0.4	2.8

Note: Cement addition estimated assuming 45% of ore requires agglomeration using cement.

13.4.2 Milling Recovery Estimates

Metallurgical testing (primarily flotation and agitated cyanidation) has shown that the DeLamar non-oxide materials respond well to flotation at a moderate grind size (150 microns) for recovery of gold and silver to a flotation concentrate. The resulting flotation concentrate responds well to cyanide leaching after very fine regrinding (20 microns) for recovery of contained silver. Some gold is also recovered by cyanide leaching of the reground flotation concentrate, but those recoveries generally are low. Mineralogical examination and metallurgical testing have shown that these materials contain significant amounts of gold that are locked in sulfide mineral particles, which require oxidative pretreatment of sulfide minerals for liberation of gold before high cyanidation gold recoveries can be obtained. Expected recoveries from the DeLamar non-oxide mineralization in the planned mill circuit, consisting of grinding, flotation concentrate regrinding and cyanide leach, range from 28% to 39% for gold and from 64% to 87% for silver. Estimated mill recoveries and reagent consumptions are presented in Table 13.8 and Table 13.9.

Table 13-8 DeLamar PFS Non-Oxide Mill Recovery Estimates

DeLamar Non-Oxide Recoveries	
Overall Gold Recoveries, %	
Sullivan Gulch	38
Glen Silver	28
All Others (DLM)	39
Overall Silver Recoveries, %	
Sullivan Gulch	73
Glen Silver	64
All Others (DLM)	87
Flotation (80% -150µm) with Concentrate Regrind/Cyanide Leach (80% -20µm)	

DLM is DeLamar

Table 13-9 DeLamar PFS Non-Oxide Reagent Estimates

DeLamar Non-Oxide Reagent Estimates		
Reagents - flotation	**Value**	**Unit**
$CuSO_4*5H_2O$	0.10	kg/t ore
Na_2CO_3 (sodium carbonate pH modifier)[1)]	0.25	kg/t ore
Potassium Amyl Xanthate (PAX collector)	0.025	kg/t ore
AERO 208 (dithiophosphate collector)	0.05	kg/t ore
MIBC	0.01	kg/t ore
Na_2SiO_3 (sodium silicate dispersant)[2)]	0.025	kg/t ore
Reagents - Flotation Concentrate Cyanidation		
	kg/t ore	
Area	NaCN	CaO
Sullivan Gulch	0.86	0.5
Glen Silver	1.25	0.7
DeLamar - all other areas	0.55	0.3
Flotation (80%-150µm) with Concentrate Regrind/Cyanide Leach (80%-20µm)		
1) Sodium carbonate addition estimated assuming 1.0 kg/mt required by 25% of DeLamar non-oxide feed.		
2) Sodium silicate addition estimated assuming 0.25 kg/mt required by 10% of DeLamar non-oxide feed.		

Metallurgical testing has shown that the non-oxide mineralization from the Florida Mountain deposit responds well to upgrading by flotation at a moderate grind size (150 microns) and cyanidation gold and silver recoveries from the resulting concentrates can be maximized by very fine regrinding (20 microns). In contrast to the DeLamar non-oxide materials, oxidative pretreatment of contained sulfide minerals is not required to achieve high cyanidation gold recoveries from the Florida Mountain non-oxide feeds. Recoveries expected from the Florida Mountain non-oxide mineralization in the planned mill circuit vary with feed grade, but generally are high, with maximum recoveries of 87% gold and 77% silver. Estimated overall recoveries and reagent consumptions are shown in Table 13.10 and Table 13.11, respectively.

Table 13-10 Florida Mountain PFS Non-Oxide Overall Mill Recoveries

Overall Gold Recovery, %	
Head Grade	Combined flot/regrind/CN Au Rec
For head grade > 0.65 g Au/t	87%
For head grade less than or equal to 0.65 g Au/t	Eq 1: % Au = (14.562*ln(GPTAU)+102.21)*0.91
Overall Silver Recovery, %	
Head Grade	Combined flot/regrind/CN Ag Rec
For head grade > 20.9 g Ag/t	77%
For head grade less than or equla to 20.9 gAg/t	Eq 2: % Ag = (13.021*ln(GPTAG)+48.447)*0.88
Flotation (80% -150µm) with Concentrate Regrind/Cyanide Leach (80% -20µm)	
Where GPTAU is the mill feed grade in gAu/mt. GPTAG is the mill feed grade in gAg/mt.	

Table 13-11 Florida Mountain PFS Non-Oxide Overall Reagent Estimates

Reagents - Flotation	Value	Unit
$CuSO_4*5H_2O$	0.10	kg/mt ore
Sodium Carbonate	0.15	kg/t ore
Potassium Amyl Xanthate (PAX collector)	0.025	kg/t ore
AERO 208 (dithiophosphate collector)	0.05	kg/t ore
MIBC	0.01	kg/t ore
Reagents - Flotation Concentrate Cyanidation	**Value**	**Unit**
NaCN	0.26	kg/t ore
CaO	0.10	kg/t ore
Flotation (80% -150µm) with Concentrate Regrind/Cyanide Leach (80% -20µm)		

13.5 Post PFS Historic Backfill and Waste Dump Testing

In October of 2022, Integra started a drilling program using sonic and reverse-circulation drills on the historic backfill and waste dumps at DeLamar and Florida Mountain. Material from the drill program was sampled and used for metallurgical testing. The historic backfill and waste dump material was classified into six areas: North DeLamar Backfill, Sommercamp Backfill, Waste Dump #1, Waste Dump #2, Jacob's Gulch Dump, and Tip Top Backfill. The Jacob's Gulch Dump and Tip Top Backfill are comprised of material removed from the Florida Mountain deposit; while the other four areas are comprised of material removed from the DeLamar deposit.

A variability bottle roll program is in progress for each of the six areas. Results as of August 25th, 2023 are summarized below in Table 13.12 and Table 13.13 (McPartland, 2023b).

Table 13-12 Historic Backfill and Dump Preliminary BRT Gold Recovery (%) Results

Area	Test Count	Average	Max	Min
North DeLamar Backfill	50	69.0	90.2	28.6
Sommercamp Backfill	14	70.3	90.0	56.7
Waste Dump 1	52	78.4	93.8	54.7
Waste Dump 2	34	78.1	92.9	40.0
Jacob's Gulch Dump	46	89.4	97.1	75.0
Tip Top Backfill	26	90.9	95.5	78.9

Table 13-13 Historic Backfill and Dump Preliminary BRT Silver Recovery Results

Area	Test Count	Average	Max	Min
North DeLamar Backfill	50	36.0	52.6	21.1
Sommercamp Backfill	14	41.8	65.0	25.0
Waste Dump 1	52	39.6	74.4	18.6
Waste Dump 2	34	40.3	68.8	27.6
Jacob's Gulch Dump	46	47.7	70.0	25.0
Tip Top Backfill	26	51.3	68.8	30.0

Bottle roll cyanide consumptions generally were low (<0.5 kg NaCN/mt). Hydrated lime requirements were variable and averaged 3 kg/mt.

Additional variability bottle roll testing is currently being completed. Column leach testing for each of the areas is also being started. One P_{80} - ½" column test is being completed for each one of the six areas. Column feed size was based on the heap leach crush size designated in the PFS.

13.6 Summary Statement

Mr. McPartland has reviewed the historical metallurgical studies and the metallurgical studies conducted during 2018 through 2023 and concludes that the samples used during those studies 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 portions of the deposit, particularly those displaying high degrees of variability in metallurgical response, 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 the potential economic extraction.

The oxide and mixed materials from both the DeLamar and Florida Mountain deposits can be processed by heap-leach cyanidation. Oxide materials from both deposits are expected to give relatively high (75% - 89%) heap-leach gold recoveries, which are not particularly sensitive to feed size. Mixed materials from both deposits give lower and more variable gold recoveries (45% - 77% at a 12.7mm crush size) (0.5-inch) which are more sensitive to feed size. Heap-leach silver recoveries expected from the oxide and mixed materials are significantly lower, more variable (11% - 74% at a 12.7mm crush size), and more sensitive to feed size. A relatively fine (80% -12.7mm) (-0.5-inch) feed size was selected for heap leaching the DeLamar and Florida Mountain oxide and mixed materials to maximize silver recovery, and in the case of the mixed materials, to maximize gold recovery. Some of the DeLamar oxide and mixed materials contain elevated levels of clay and will require agglomeration pretreatment before heap leaching. None of the Florida Mountain materials require agglomeration. Non-oxide materials from both deposits generally are not expected to be amenable to heap-leach cyanidation.

Non-oxide materials from the DeLamar deposit generally give low but highly variable cyanidation gold recoveries, even after very fine grinding. These materials contain significant amounts of gold that are locked in sulfide mineral particles. For these materials, oxidative pretreatment (such as the Albion process) of sulfide minerals is required for liberation of gold before high cyanidation recoveries can be obtained. In contrast, silver contained in the DeLamar non-oxide materials generally can be liberated to a large degree by very fine grinding (-20 microns). Cyanidation silver recoveries after very fine grinding generally are relatively high (approximately 80%). The DeLamar non-oxide materials respond well to flotation at a moderate grind size (150 microns) (100 mesh). The resulting flotation concentrate responds well to cyanide leaching after very fine regrinding for recovery of contained silver. Some gold is also recovered by cyanide leaching of the reground flotation concentrate, but those recoveries generally are low.

Non-oxide materials from the Florida Mountain deposit are amenable to cyanidation treatment at moderately fine grind sizes (75 microns and finer) (200 mesh and finer) and give relatively high gold recoveries (generally >80%) and moderate silver recoveries (generally >50%). These recoveries can be improved by finer grinding. These materials respond well to upgrading by flotation at a moderate grind size (150 microns) and cyanidation gold and silver recoveries from the resulting concentrates can be maximized by very fine regrinding (20 microns). In contrast to the DeLamar non-oxide materials, oxidative pretreatment of contained sulfide minerals is not required to achieve high cyanidation gold recoveries from the Florida Mountain non-oxide feeds.

Preliminary heap leach cyanidation (bottle roll) testing has demonstrated that the historical dump and backfill materials should be amenable to heap leach cyanidation processing, using the same process flowsheet as planned for the in-situ oxide and mixed materials. Recoveries are expected to be variable, but generally fairly high (>70%). Silver recoveries are expected to be variable and generally substantially lower (<50%). Cyanide consumptions are expected to be low to moderate. Lime requirements are expected to be variable. Agglomeration pretreatment will likely be required for some of the backfill and dump materials.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The updated mineral resource estimations for the DeLamar project, which include resources of the DeLamar and Florida Mountain areas, were completed for public disclosure in accordance with the guidelines of NI 43-101. The mineral resources were estimated under the supervision of Mr. 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 August 25, 2023, the date the final economic parameters that were ultimately applied to the resource pit optimizations were defined by RESPEC and Integra. Of note, for the first time the current resources summarized below include stockpile resources that are derived from materials mined but not processed during the historical open-pit operations at 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 as well as the data derived from the exploration programs of the various historical operators discussed in Section 10.0. This information, most importantly including the data derived from RC, conventional rotary, diamond-core, and sonic 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 by Integra.

14.2.1 Drill-Hole Data

The historical project data utilized mine-grid coordinates, a local grid system in Imperial units developed in the early 1970s and used throughout the life of the DeLamar project 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 RESPEC. All project down-hole drill depths, assays, and geologic logging intervals were then converted into meters and grams-per-tonne.

As discussed in Section 10.7, drill intervals identified as having significant sample quality issues, including poor sample recoveries and down-hole contamination, were excluded from use in the resource estimation. In addition, sample intervals of colluvial materials 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 both the DeLamar and Florida Mountain database. The excluded colluvial sample intervals are commonly mineralized, especially at Florida Mountain where significantly mineralized colluvium was frequently intersected in the top few meters of drill holes.
After verifying the historical data, RESPEC constructed resource databases for the DeLamar and Florida Mountain areas. Historical drill-hole gold and silver values used in the current resource estimations were prioritized as follows: fire assays by outside labs were given top priority, followed by fire assays by the on-site mine lab, with mine-lab AA gold analyses selected only in cases where no other data were available. Certain low-precision gold analyses and all mine-lab AA silver assays were identified, flagged in the resource databases, and not used in the estimation of the project resources (see Section 12.0).

14.2.2 Topography

Integra provided RESPEC 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. These current topographic surfaces reflect post-mining reclamation, including re-contouring of waste dumps and the partial backfilling of many of the open pits.

Integra also provided original historical paper plots of final post-mining topographies of the historical open pits at both the Florida Mountain and DeLamar areas. RESPEC used these paper plan maps to create digital ‘as-mined’ topographic surfaces that encompass 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 and Integra drilling through backfilled areas, Mr. Gustin believes the modeled as-mined surfaces reasonably represent the volumes mined during the historical open-pit operations.

14.3 Modeling of Historical Underground Workings

Integra provided RESPEC 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. RESPEC 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.

14.4 Geological Modeling

Integra completed digital lithological and structural interpretations on sets of cross sections that span the extents of the Florida Mountain and DeLamar resource areas. These cross sections were used as the base for modeling the gold and silver mineral domains discussed in Section 14.9.1. 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 RESPEC, were represented as three-dimensional wireframe surfaces that served as guides for the detailed modeling of the gold and silver mineralization as part of the estimation of the project mineral resources.

14.5 Deposit Geology Pertinent to Resource Modeling

The DeLamar area mineralization is predominantly influenced by moderate- to high-angle zones of higher-grade mineralization and associated much larger bodies that halo the higher-grades. In some areas, moderately dipping mineralization flattens upwards to the northeast, although significant portions of these more shallowly dipping zones of mineralization were mined in the historical operations. The mineralization is overwhelmingly hosted in the felsic volcanic units that lie above the lower basalt and below the banded rhyolite. While only minor mineralization has been drilled within the lower basalt, low-grade mineralization does occur locally within banded rhyolite that lies in the uppermost portions of the felsic package.

At Florida Mountain, the gold and silver mineralization drilled to date also occurs primarily within felsic volcanic units, which in this area overlie Cretaceous granodiorite. The granodiorite hosts most of the high-grade veins that were the focus of the historical underground mining at Florida Mountain, although the Trade Dollar vein in particular was mined into the felsic package, locally through to the present-day surface. The Florida Mountain mineralization that comprises the current resources occurs along multiple, broad, north- to north-northwest-striking zones straddling the contacts of rhyolitic intrusive necks that intrude the felsic volcanic units. 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, and taken as a whole, these zones formed bulk-mineable bodies. These zones blossom outwards from the intrusive rhyolite necks upwards towards the surface and collapse inward towards the rhyolite necks at deeper levels in the felsic volcanic package. The continuations of the mineralization into higher-grade, more discrete zones at depth, especially into the granodiorite, are presently being explored as potential underground mineable targets.

Mr. Gustin reviewed the distribution of the silver mineralization intersected in drilling carefully, especially in the silver-rich DeLamar area, to discern the presence (or absence) of potential supergene-enriched zones that would be relevant to the resource modeling. Only a few limited areas were found that are suggestive of possible supergene enrichment, but the evidence is 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.

14.6 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 (180 feet) 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 (~6,053 feet), considerably deeper than the level of oxidation. These statements presumably applied only to the two deposit areas that were the subject of the historical feasibility study. A later mine document reported a water table depth of 1,810 meters (~5,938 feet) 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 RESPEC 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 (5,971 and 6,250 feet), respectively.

14.7 Oxidation Modeling

Integra completed comprehensive logging of oxidation of the historical RC and rotary holes using the chipboards present at the project site. This information was then combined with oxidation logging of Integra RC chips and drill core and added to the project databases. While earlier resource modeling relied almost exclusively on these visual logging codes, there is now sufficient Integra drilling to incorporate chemical analyses pertinent to oxidation state into the modeling. The chemical data include cyanide-leach analyses of drill-sample pulps, ICP sulfur data, and limited LECO sulfur speciation data. The cyanide-leach analyses were used to calculate cyanide-leach-gold-to-fire-assay-gold ("CN/FA") ratios.

RESPEC created wireframe solids of oxide and non-oxide zones based on the visual logging of oxidation state and the chemical data summarized above, and the resultant solids were used to code the DeLamar and Florida Mountain block models. Model blocks lying between those coded as oxide or non-oxide, which are comprised of oxide, partially oxidized, and non-oxide materials that lack continuity to be modeled separately, were assigned a "mixed" code.

Despite the addition of the chemical data with every new hole drilled by Integra, the visual logging codes are still the dominant input to the modeling of oxidation state. In order to evaluate the accuracy of the logging, RESPEC requested Integra to send a large batch of Integra drill-sample pulps for cyanide-leach assay; these sample pulps were from holes drilled prior to the now routine cyanide-leach analyses of all samples within mineralized zones, and they therefore lacked cyanide-leach assays. The incorporation of this new cyanide-leach data led, in general, to small expansions of the previously modeled mixed zones at the expense of previously modeled non-oxide zones. While the impacts were small, they indicate that the modeling of non-oxide materials at the mixed / non-oxide boundary will tend to slightly overstate the non-oxide materials in areas lacking chemical data pertinent to the assignment of oxidation state.

The coding of oxidation state in the DeLamar and Florida Mountain models determines the potential processing option for each block in each model, and therefore the resource cutoff grade that applies to each block (discussed further below).

The CN/FA ratios indicate that there are areas of mixed materials within the modeled non-oxide zone at the Sullivan Gulch portion of the DeLamar area, but there are insufficient CN/FA ratio assays to model these zones confidently. These unmodeled mixed materials likely are related to fault zones through which oxygenated meteoric waters percolated and partially oxidized otherwise non-oxide zones.

14.8 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 f\or 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.

RESPEC 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. A total of 12 historical SG determinations from Florida Mountain drill core compiled by RESPEC average 2.41. 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.

Integra routinely measured the SG of selected samples of its drill core using the water immersion method. The resulting SG datasets, comprised of over 1,500 determinations from the DeLamar resource area and more than 3,400 at Florida Mountain, were statistically evaluated by the presence or absence of gold and/or silver mineralization, oxidation state, and lithology. Table 14-1 and Table 14-2 show the SG values used in the resource estimations of the DeLamar and Florida Mountain deposits (see Section 14.9.1). Mineralized SG samples are those lying within the gold and/or silver mineral domains that constrain the resource estimations.

Table 14-1 DeLamar Area Specific Gravity Values Assigned to Resource Model

Type	Lithology	Oxide	Mixed	Non-Oxide	No Ox Coding
Mineralized	All	2.34	2.50	2.55	-
Unmineralized	Colluvium	1.70	1.70	1.70	-
	Banded Rhyolite	2.22	2.22	2.22	-
	Sediments	1.84	1.84	2.02	1.84
	Porphyritic Rhyolite	2.20	2.35	2.35	2.28
	Tuff Breccia	2.15	2.15	2.22	-
	Quartz Latite	2.45	2.45	2.45	-
	Porphyritic Latite	2.38	2.38	2.38	-
	Lower Basalt	2.26	2.26	2.52	2.26
	unassigned	-	-	-	2.34

Table 14-2 Florida Mountain Area Specific Gravity Values Assigned to Resource Model

Type	Lithology	Oxide	Mixed	Non-Oxide	No Ox Coding
Mineralized	All	2.42	2.48	2.48	-
Unmineralized	Stockpiles	1.75	1.75	1.75	-
	Rhyolite 1	2.33	2.33	2.33	-
	Rhyolite 2	2.30	2.46	2.49	-
	Rhyolite 3	2.48	2.48	2.48	-
	Rhyolite 4	2.41	2.45	2.45	-
	Sediments-Tuff	2.48	2.48	2.48	-
	Tuff	2.43	2.43	2.36	-
	Quartz Latite	2.38	2.46	2.46	-
	Lower Basalt	2.60	2.60	2.60	-
	Granodiorite	2.59	2.59	2.59	-
	unassigned	2.40	2.45	2.45	-

While the oxidation solids encompass all modeled mineralized areas, not all unmineralized areas are covered. The lithologic solids also do not cover the full extents of either the DeLamar or block models. Where there is sufficient data, SG values tend to increase from oxide to mixed to non-oxide zones, which is at least in part due to decreasing effects of weathering (oxidation). The raw data also suggests SG may increase slightly as metal grades increase, but the data quantity is insufficient to be confident that this is the case.

Specific gravity values of 1.70 and 1.75 were assigned to backfill/stockpile materials at the DeLamar and Florida Mountain areas, respectively. The slightly higher SG at Florida Mountain is to account for the higher rock SGs.

14.9 DeLamar Area In Situ Gold and Silver Modeling

14.9.1 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 the DeLamar project, the natural gold and silver populations were first identified on population-distribution graphs that plot the gold-grade and silver-grade distributions of all drill-hole assays at, in this case, the DeLamar area. 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 (98.4-foot) 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, were critical to the high-confidence modeling of the mineral domains.

The low-grade gold and silver domains ("100" domains) generally encompass relatively extensive southwest-dipping bodies lying within the various felsic volcanic units that lie between the banded rhyolite and the lower basalt. Relatively restricted, moderately to steeply dipping zones of mineralization in the mid-grade and higher-grade domains (domains "200" and "300", respectively) occur within the broad extents of the lower-grade mineralization.

In areas where all volcanic units from the lower basalt to the flow-banded rhyolite were preserved, the higher-angle mineralization often bends into approximate parallelism with the basal contact of the flow-banded rhyolite. The domain 200 and 300 mineralization then extends laterally in a northeasterly direction along or close to this contract. 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 lower-angle mineralization were present prior to mining. In this case, the higher 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 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 (0.75 miles), and an elevation range of 570 meters (1,870 feet). The Milestone portion of the DeLamar mineralization, which lies about three-quarters of a kilometer (0.47 miles) northwest of the northwesternmost extents of the main DeLamar area, adds an additional 640 meters (2,100 feet) of strike to the resource modeling.

Cross sections showing examples of the modeled in-situ gold and silver mineral domains for the Sullivan Gulch and Glen Silver – South Wahl areas of the resources are shown in Figure 14.1 through Figure 14.4.

The final cross-sectional gold and silver mineral-domain polygons were projected horizontally to the drill data within each sectional window, and these three-dimensional polygons were then sliced horizontally at six-meter (19.68-foot) elevation intervals that match the mid-block elevations of the resource block model. The slices were used to create a new set of mineral-domain polygons for both gold and silver on level plans at six-meter (19.68-foot) vertical spacings. Level plans were used due to the predominance of moderately to steeply dipping mineralization, especially in medium- and high-grade domains.

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 mineral-domain polygons serve to rectify the gold and silver domains to the drill-hole data at the scale of the block model.

Figure 14-1 Cross Section 1230 NW Showing Gold Domains at Sullivan Gulch

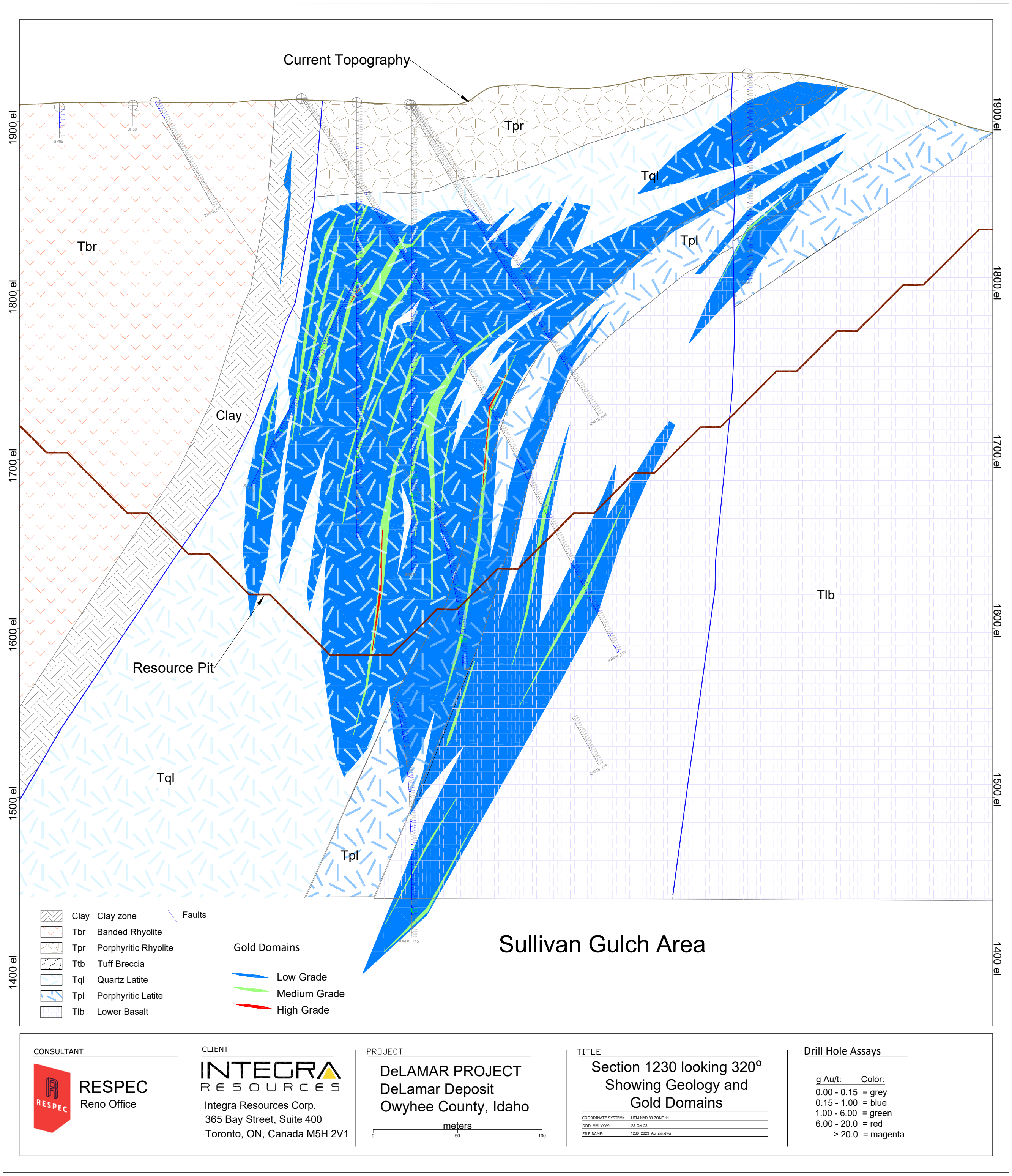


Figure 14-2 Cross Section 1230 NW Showing Silver Domain at Sullivan Gulch

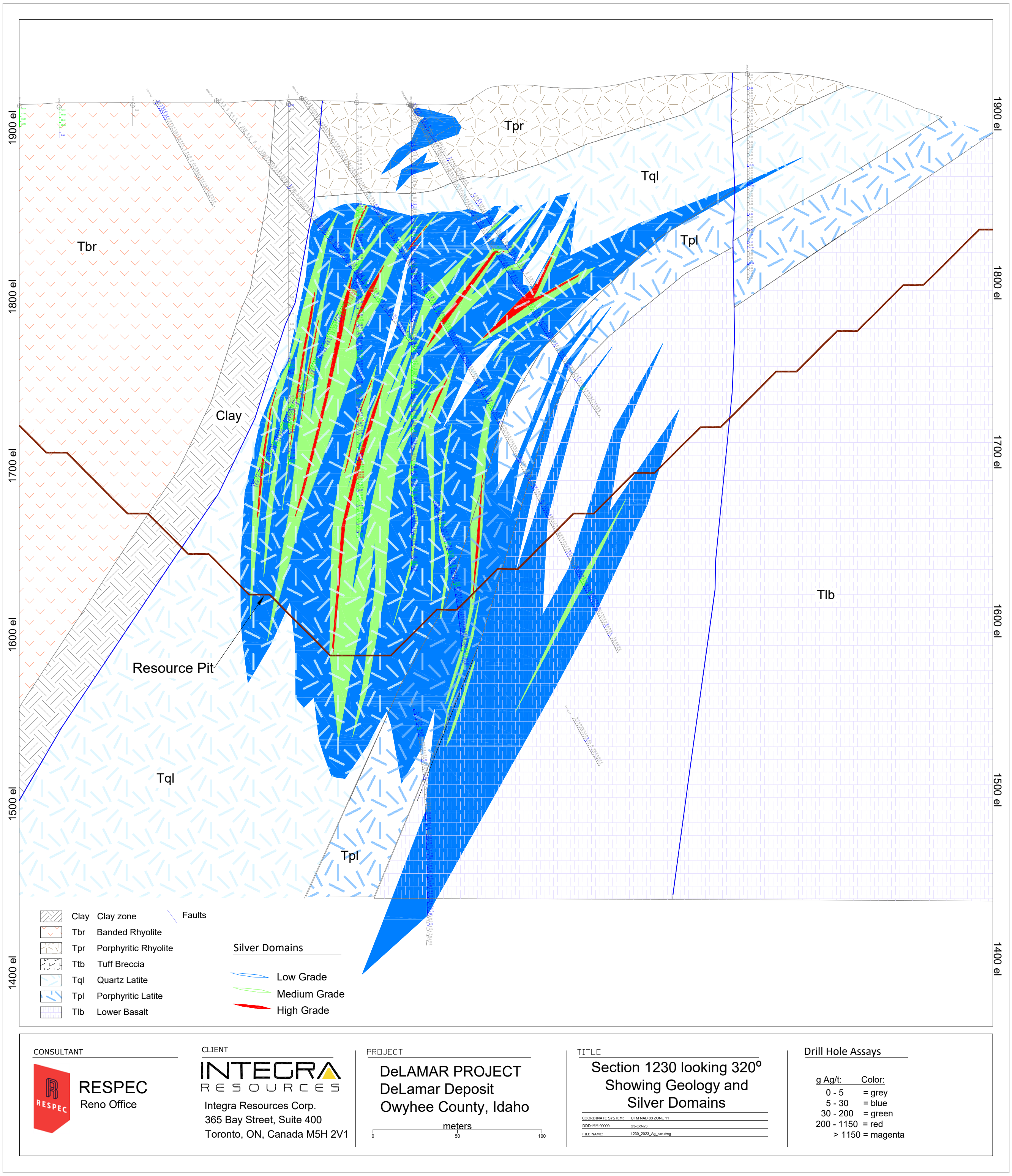


Figure 14-3 Cross Section 2790 NW Showing Gold Domains at Glen Silver and South Wahl

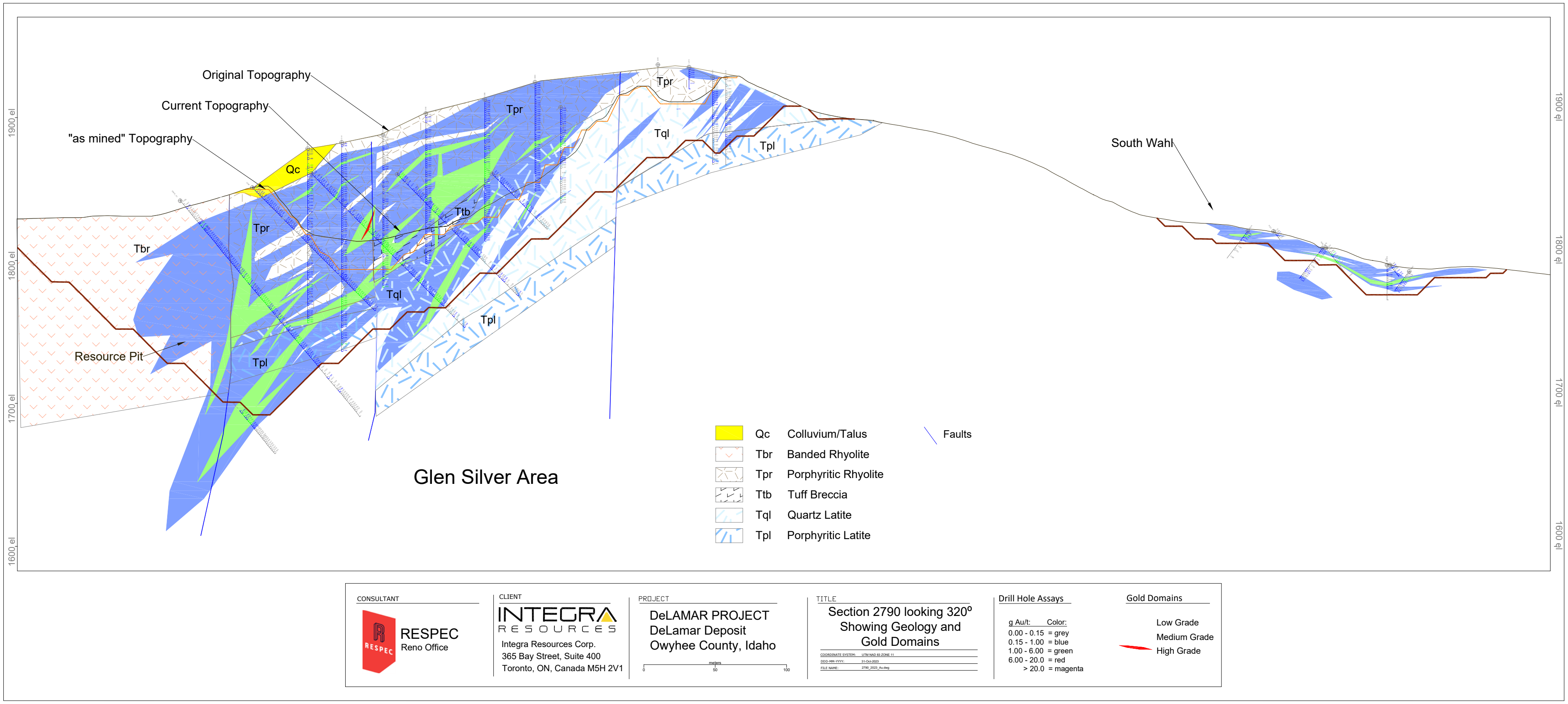
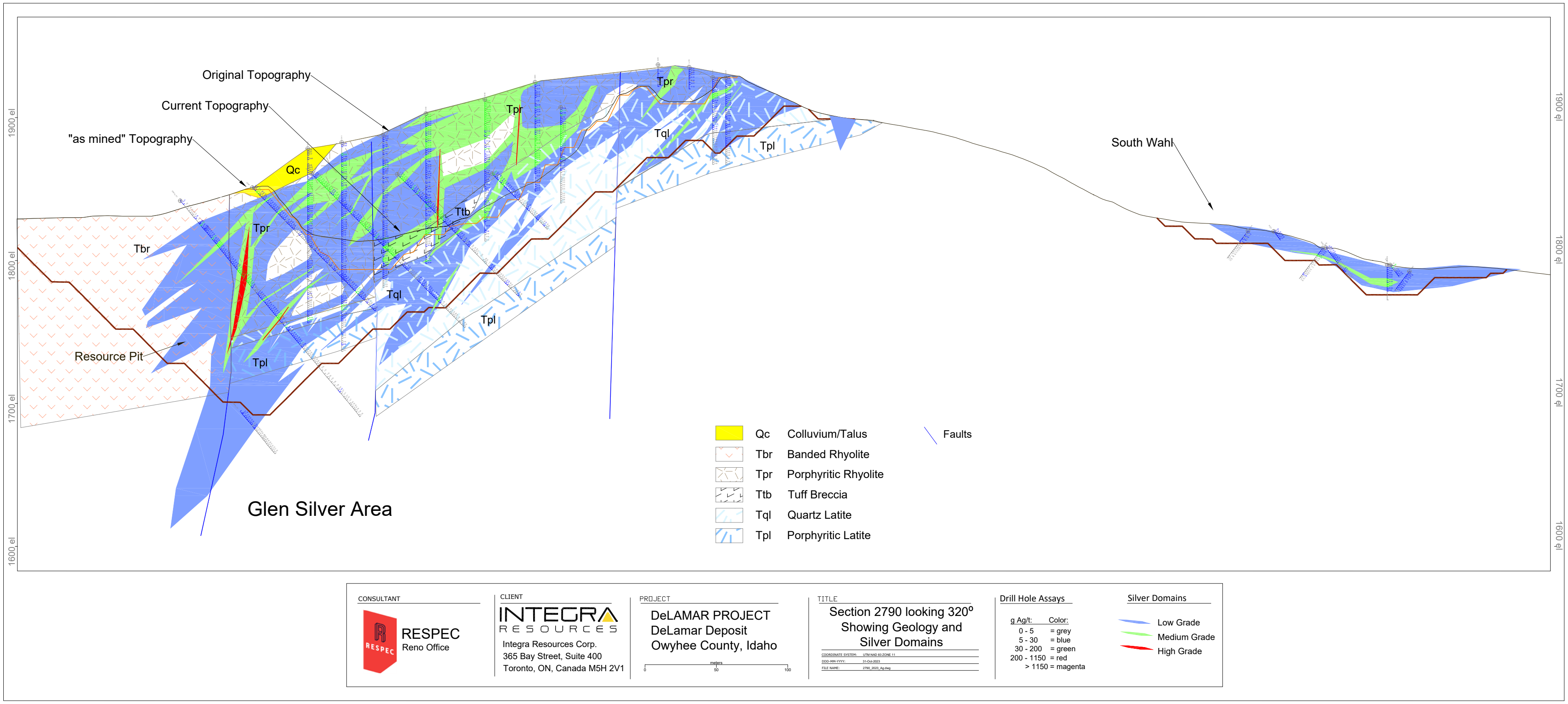


Figure 14-4 Cross Section 2790 NW Showing Silver Domains at Glen Silver and South Wahl



14.9.2 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 low-, mid-, 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. Table 14-4 shows the gold and silver assay cap applied to each of the domains.

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	1	74 (<1%)	150	8 (<1%)
100	2.5	36 (<1%)	125	15 (<1%)
200	6	14 (<1%)	200	59 (<1%)
300	40	8 (3.0%)	1,325	29 (1.5%)

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	47,890	0.07	0.04	0.46	6.45	0.00	102.86
	Au Cap	47,890	0.07	0.04	0.10	1.41	0.00	1.00
100	Au	35,113	0.36	0.30	0.26	0.71	0.00	9.70
	Au Cap	35,113	0.36	0.30	0.23	0.64	0.00	2.50
200	Au	4,437	1.70	1.34	1.31	0.77	0.00	46.29
	Au Cap	4,437	1.68	1.34	1.06	0.63	0.00	6.00
300	Au	269	14.58	9.19	26.51	1.82	0.21	368.64
	Au Cap	269	12.25	9.19	8.25	0.67	0.21	40.00
100+200+300	*Au*	*39,819*	0.61	0.34	2.54	4.20	0.00	368.64
	Au Cap	*39,819*	0.59	0.34	1.31	2.24	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	Au	34,194	2.34	0.69	9.40	4.02	0.00	790.97
	Au Cap	34,194	2.29	0.69	7.24	3.16	0.00	150.00
100	Au	27,875	12.75	10.29	10.21	0.80	0.00	236.87
	Au Cap	27,875	12.73	10.29	9.96	0.78	0.00	125.00
200	Au	13,273	57.29	47.31	41.54	0.73	0.00	2402.40
	Au Cap	13,273	56.81	47.31	34.53	0.61	0.00	200.00
300	Au	1,930	308.08	212.50	409.48	1.33	6.86	14054.00
	Au Cap	1,930	288.60	212.50	231.55	0.80	6.86	1325.00
100+200+300	*Au*	*43,078*	39.29	17.14	107.71	2.74	0.00	14054.00
	Au Cap	*43,078*	38.28	17.14	77.60	2.03	0.00	1325.00

In addition to the assay caps, restrictions to the search distances of higher-grade composites within 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	42,352	0.05	0.03	0.04	0.85	0.00	0.80
100	19,197	0.36	0.31	0.19	0.54	0.00	2.50
200	2,835	1.68	1.41	0.90	0.54	0.03	6.00
300	203	12.25	9.62	7.38	0.60	0.21	40.00
100+200+300	*22,235*	*0.59*	*0.34*	*1.26*	*2.14*	*0.00*	*40.00*

Table 14-8 Descriptive Statistics of DeLamar Area Silver Composites

Domain	Count	Mean (g Au/t)	Median (g Au/t)	Std. Dev.	CV	Min. (g Au/t)	Max. (g Au/t)
0	42,352	0.05	0.03	0.04	0.85	0.00	0.80
100	15,931	12.73	10.63	8.50	0.67	0.00	125.00
200	7,924	56.81	49.03	29.37	0.52	0.00	200.00
300	1,257	288.60	222.34	206.82	0.72	6.86	1325.00
100+200+300	*25,112*	*38.28*	*17.14*	*73.70*	*1.93*	*0.00*	*1325.00*

14.9.3 Block Model Coding

The six-meter-spaced level-plan and long-sectional mineral-domain polygons were used to code a model comprised of 6 x 6 x 6-meter (19.68 x 19.68 x 19.68-foot) blocks. The model is rotated to a bearing of 320°, which is consistent with orientation of the cross sections. The percentage volume of each mineral domain (the "partial percentages") for both gold and silver, as coded directly by the level plans, is stored within each block, as is the volume percentage of the block that lies outside of the modeled domains for both gold and silver.

Two topographic surfaces were used to code the block model: the as-mined and present-day surfaces discussed in 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 14.7 were used to code each model block as oxide, mixed, or non-oxide. The specific-gravity values were assigned to each block as discussed in Section 14.8.

14.9.4 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	20	2	12	4
Pass 1 + 2 – Doman 200 + 300 + 0	60	60	20	2	20	4
Pass 3 – Doman 100	60	60	60	1	12	4
Pass 3 – Doman 0 + 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
Au 0	>0.5 g Au/t	6 meters	1, 2, 3
Ag 300	>400 g Ag/t	35 meters	1, 2, 3
Ag 0	>45 g Ag/t	6 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 are less than that of the other grade interpolations. This was done to decrease the smearing of outlier high grades that are present within these otherwise low-grade domains.

The gold and silver mineralization commonly exhibits multiple orientations throughout the DeLamar deposit. A total of 12 unique dips and 4 strike directions were identified in the DeLamar resource area. These combine to create 13 unique orientation areas distributed throughout the model area, and each area was coded into the block model using wireframed 'estimation area' solids.

Many of the estimation areas are characterized by 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 was 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 used the secondary dip in its search ellipse to estimate blocks not estimated in Pass 1. 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 respect the drill data more closely 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 those mineral domains in the block, as well as any outside, dilutionary, domain 0 grades and partial percentages, 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.

14.9.5 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. The nearest-neighbor and ordinary-krige estimates, as well as a polygonal grade and tonnage estimate using the cross-sectional domain polygons, 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. The inverse-distance estimated grades were also evaluated on various grade-distribution plots that included assays, composites, and nearest-neighbor block grades as a check on both the global and local estimation results, which led to fine-tuning various estimation parameters. Finally, the inverse-distance grades were visually compared to the drill-hole assay data to assure that reasonable results were obtained.

14.10 Florida Mountain Area In Situ Gold and Silver Modeling

The modeling procedures employed for the Florida Mountain in situ resources were very similar to those used in the estimation of the DeLamar area resources (Section 14.9). The following summary of the Florida Mountain resource modeling is therefore discussed in less detail.

14.10.1 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 (98.4-foot) 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 RESPEC.

At Florida Mountain, a series of relatively thin, anastomosing, steeply dipping veins and breccias characterize the mid- to high-grade mineralization, modeled as domain 200 and 300, respectively. These thin veins and breccias are enveloped by mineralization modeled in the low-grade gold and silver domains. Taken as a whole, the mineralization forms what can be considered as large-scale stockwork systems that are associated with a series of intrusive rhyolite 'necks' (modeled by Integra as rhyolite 1, 2, 3, and 4). 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 general strike continuity, and many of these zones correlate with historically named vein zones. While the mineralization lacks continuity, especially at higher grades, the density of the drill data at Florida Mountain is sufficient to define and appropriately represent the discontinuous nature of the mineralization.

The main portion of the Florida Mountain mineralization was modeled over a northly strike extent of almost 1,400 meters (4,593 feet), an east-west width of up to 675 meters (2,215 feet), and an elevation range of approximately 500 meters (1,640 feet). The Keys area, which lies about 200 meters to the east of the main Florida Mountain resources, was added to the resource modeling in this update. The Keys mineralization has a north-south extent of over 200 meters and an east-west width of about 200 meters.

Cross-sections showing examples of the in-situ gold and silver mineral domains for the Florida Mountain deposit are shown in Figure 14.5 through Figure 14.8.

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 eight-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 eight-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.

Figure 14-5 Florida Mountain Cross Section 2830 N Showing Geology and Gold Domains

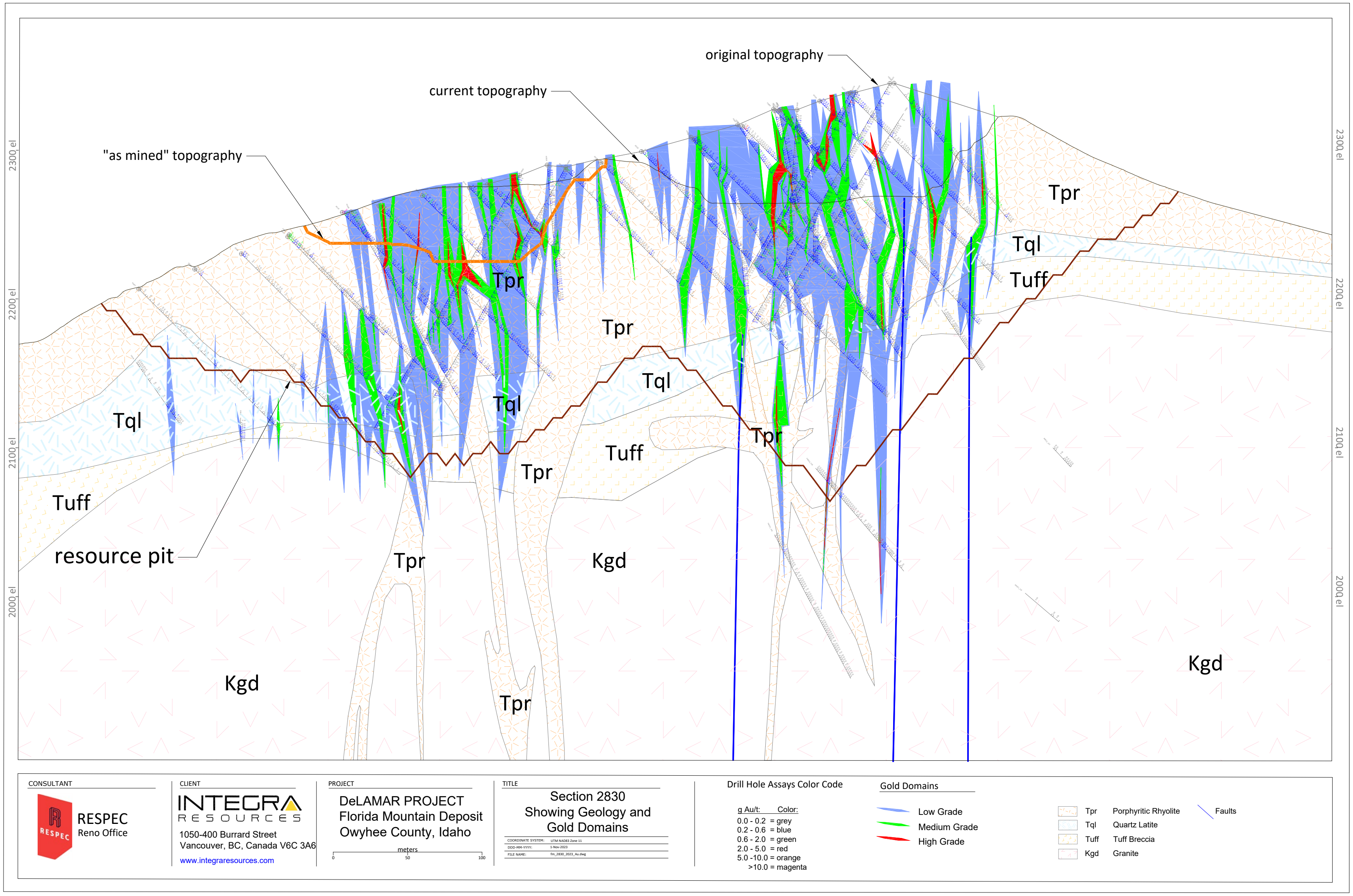


Figure 14-6 Florida Mountain Cross Section 2830 N Showing Geology and Silver Domains

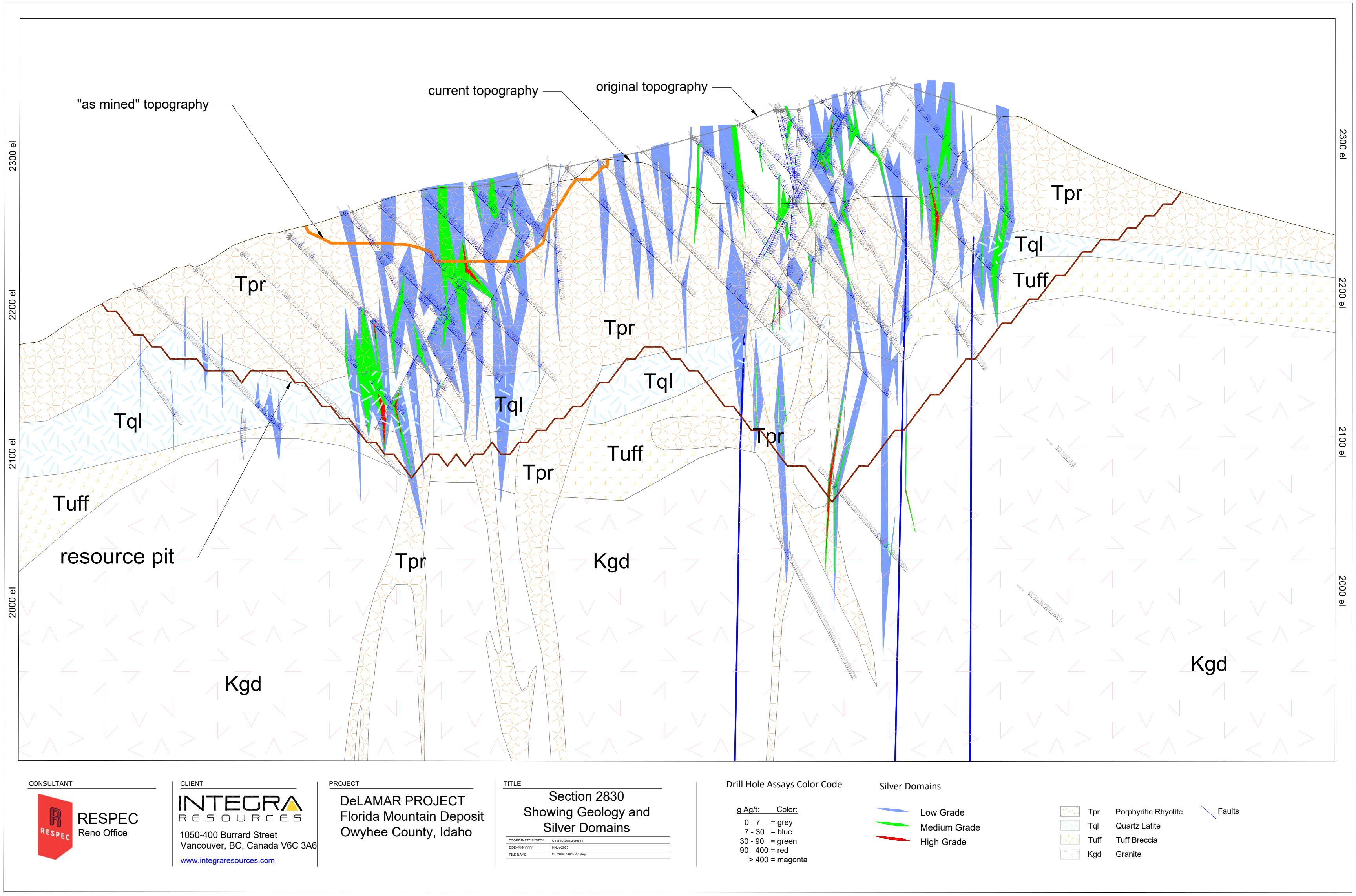


Figure 14-7 Florida Mountain and the Keys Cross Section 3280 N Showing Geology and Gold Domains

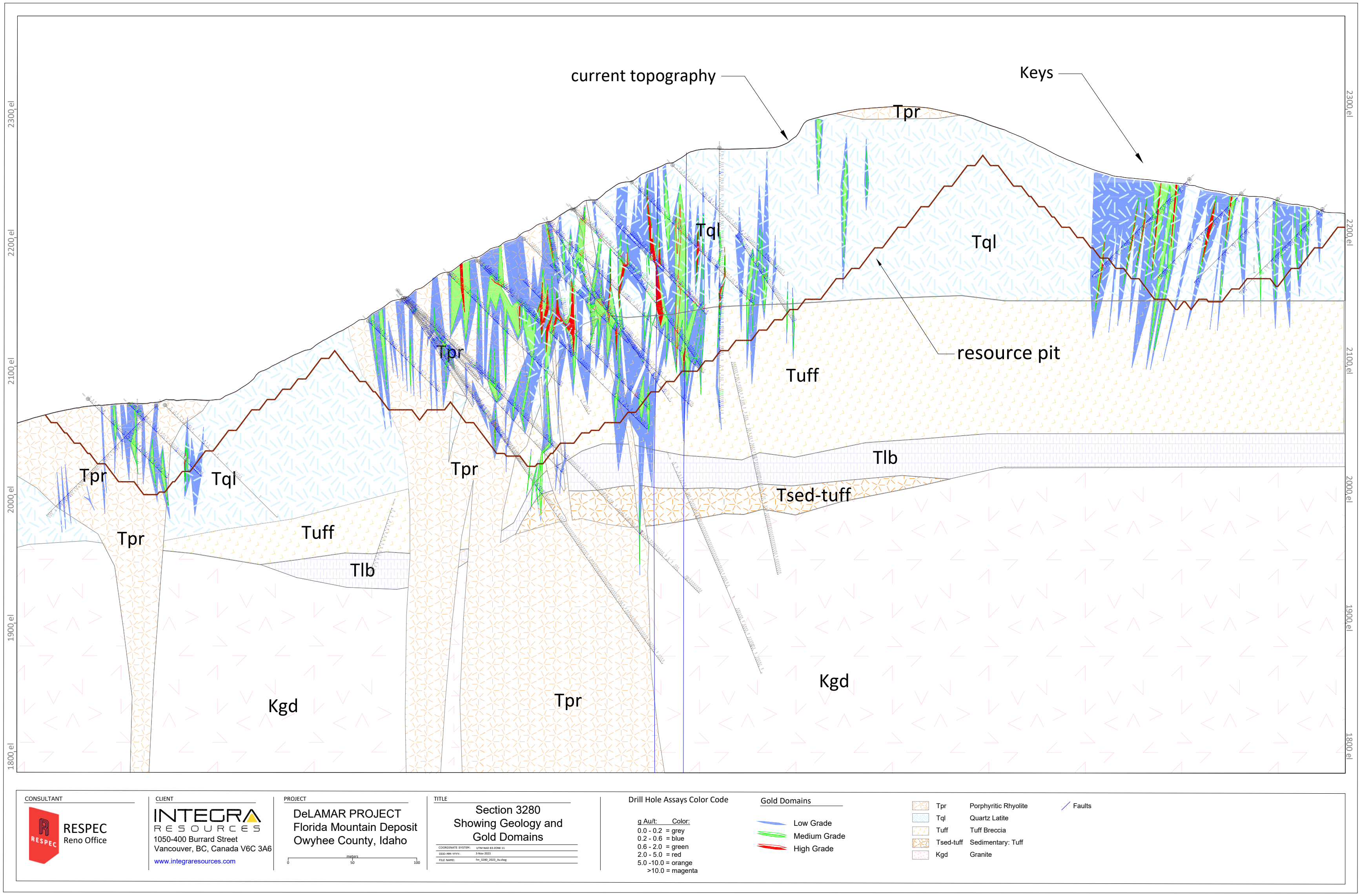
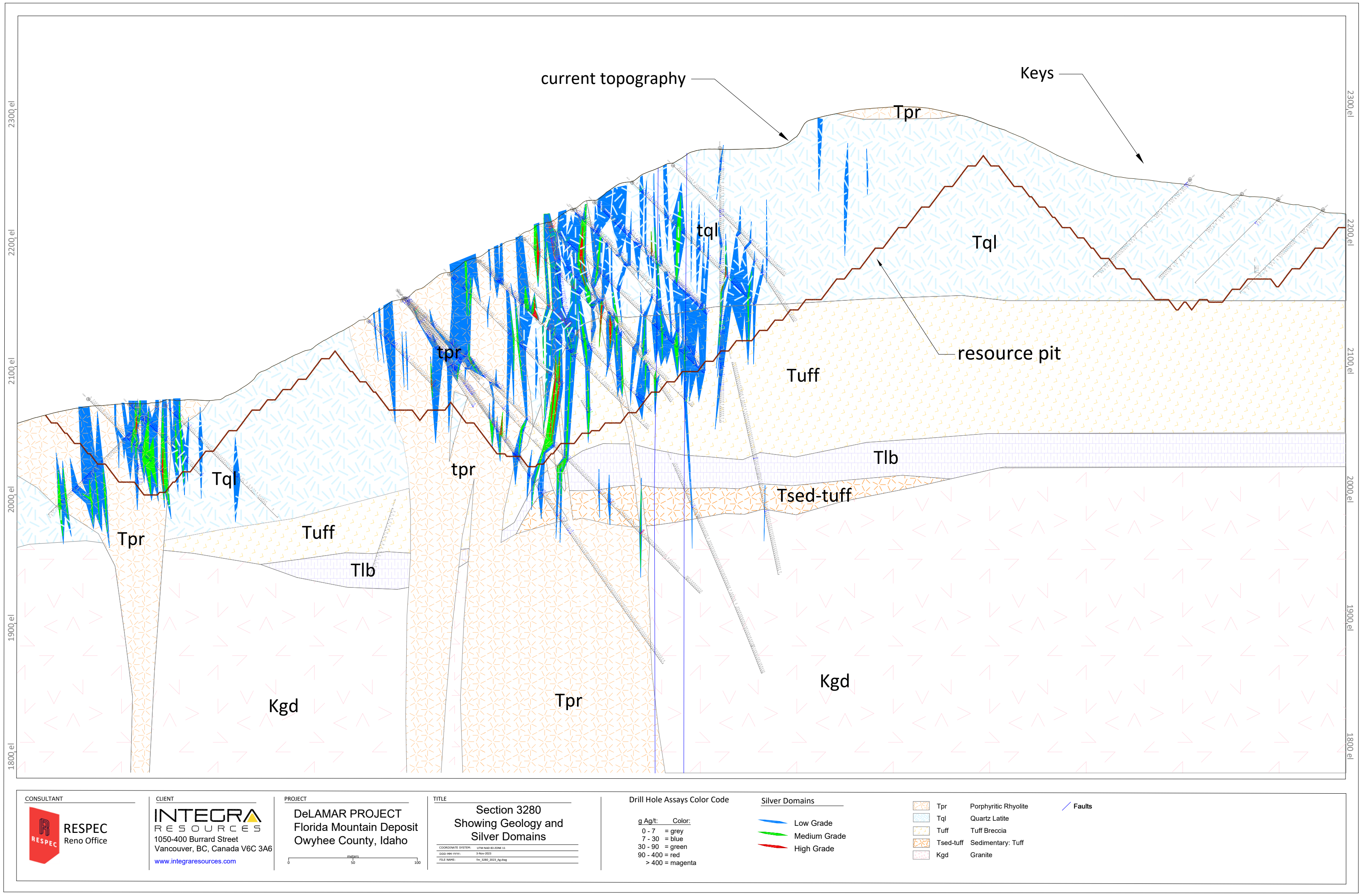


Figure 14-8 Florida Mountain and the Keys Cross Section 3280 N Showing Geology and Silver Domains



14.10.2 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 some 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	4 (<1%)	300	5 (<1%)
100	4.0	4 (<1%)	90	9 (<1%)
200	9.0	5 (<1%)	250	4 (<1%)
300	75.0	14 (<1%)	1800	14 (1.5%)

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	61,428	0.07	0.07	0.13	1.80	0.00	15.39
	Au Cap	61,428	0.07	0.07	0.10	1.43	0.00	5.00
100	Au	24,865	0.32	0.27	0.25	0.77	0.00	15.17
	Au Cap	24,865	0.32	0.27	0.21	0.66	0.00	4.00
200	Au	8,401	1.01	0.86	0.75	0.74	0.00	22.46
	Au Cap	8,401	1.01	0.86	0.71	0.71	0.00	9.00
300	Au	1,804	7.42	3.36	16.00	2.16	0.00	286.22
	Au Cap	1,804	6.87	3.36	10.83	1.58	0.00	75.00
100+200+300	*Au*	*35,070*	*0.85*	*0.34*	*3.96*	*4.67*	*0.00*	*286.22*
	Au Cap	*35,070*	*0.82*	*0.34*	*2.86*	*3.50*	*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	53,473	2.0	1.4	9.5	4.8	0.0	1865.6
	Ag Cap	53,473	2.0	1.4	4.2	2.2	0.0	300.0
100	Ag	19,478	11.7	9.6	8.1	0.7	0.0	212.0
	Ag Cap	19,478	11.7	9.6	7.7	0.7	0.0	90.0
200	Ag	3,787	45.2	39.7	34.8	0.8	0.0	1622.0
	Ag Cap	3,787	44.8	39.7	24.0	0.5	0.0	250.0
300	Ag	271	271.1	144.0	481.0	1.8	0.2	6631.0
	Ag Cap	248	247.6	144.0	283.6	1.2	0.2	1800.0
100+200+300	*Ag*	*24,201*	*26.6*	*11.4*	*106.5*	*4.0*	*0.0*	*6631.0*
	Ag Cap	*24,201*	*25.7*	*11.4*	*72.1*	*2.8*	*0.0*	*1800.0*

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	33,109	0.07	0.07	0.08	1.15	0.00	2.72
100	14,894	0.32	0.29	0.17	0.53	0.00	3.10
200	5,434	1.01	0.87	0.60	0.60	0.00	8.37
300	1,307	6.87	3.67	9.11	1.33	0.00	73.25
100+200+300	*21,635*	*0.82*	*0.35*	*2.53*	*3.09*	*0.00*	*73.25*

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	29,037	2.0	1.5	3.7	1.9	0.0	300.0
100	11,800	11.7	10.1	6.6	0.6	0.0	90.0
200	2,621	44.8	40.8	20.8	0.5	0.0	250.0
300	686	247.6	150.1	254.5	1.0	0.4	1800.0
100+200+300	*15,107*	*25.7*	*11.8*	*67.6*	*2.6*	*0.0*	*1800.0*

14.10.3 Block Model Coding

The eight-meter-spaced (26.25-foot-spaced) level-plan mineral-domain polygons were used to code a block model with a model bearing of 000° and blocks that are six meters in an east-west direction, eight meters in a north-south direction, and eight meters high. The block dimensions were increased from those used in the 2019 resource estimation and are larger than those used for the DeLamar area, reflecting conclusions derived from new geotechnical studies. 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 14.7 were used to code model blocks as oxide, mixed, or non-oxide on a block-in/block-out basis. The partial percentages of the wireframe solids of the historical underground workings 14.3) were also coded into model blocks.

Finally, the specific-gravity value for each block was coded into each model block as discussed in 14.8

14.10.4 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 respect 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	12	3	16	4
Pass 1 – Domain 200 + 300 + 0	60	60	12	3	25	4
Pass 2 – Domain 100	120	120	40	1	16	4
Pass 2 – Domain 200 + 300 + 0	120	120	40	1	25	4

Restrictions on Search Ranges

Domain	Search Restriction Threshold	Search Restriction Distance	Estimation Pass
Au 300	>15 g Au/t	20 meters	1, 2
Au 0	>0.7 g Au/t	8 meters	1, 2
Ag 300	>450 g Ag/t	35 meters	1, 2
Ag 0	>30.0 g Ag/t	6 meters	1, 2

Three estimation areas were defined for the purposes of the Florida Mountain grade interpolations, with each estimation area being characterized by a unique strike orientation (350°, 345°, and 000°) and a vertical dip.

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 those mineral domains in the block, as well as the outside, dilutionary, domain 0 grades and partial percentages, 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.

14.10.5 Model Checks

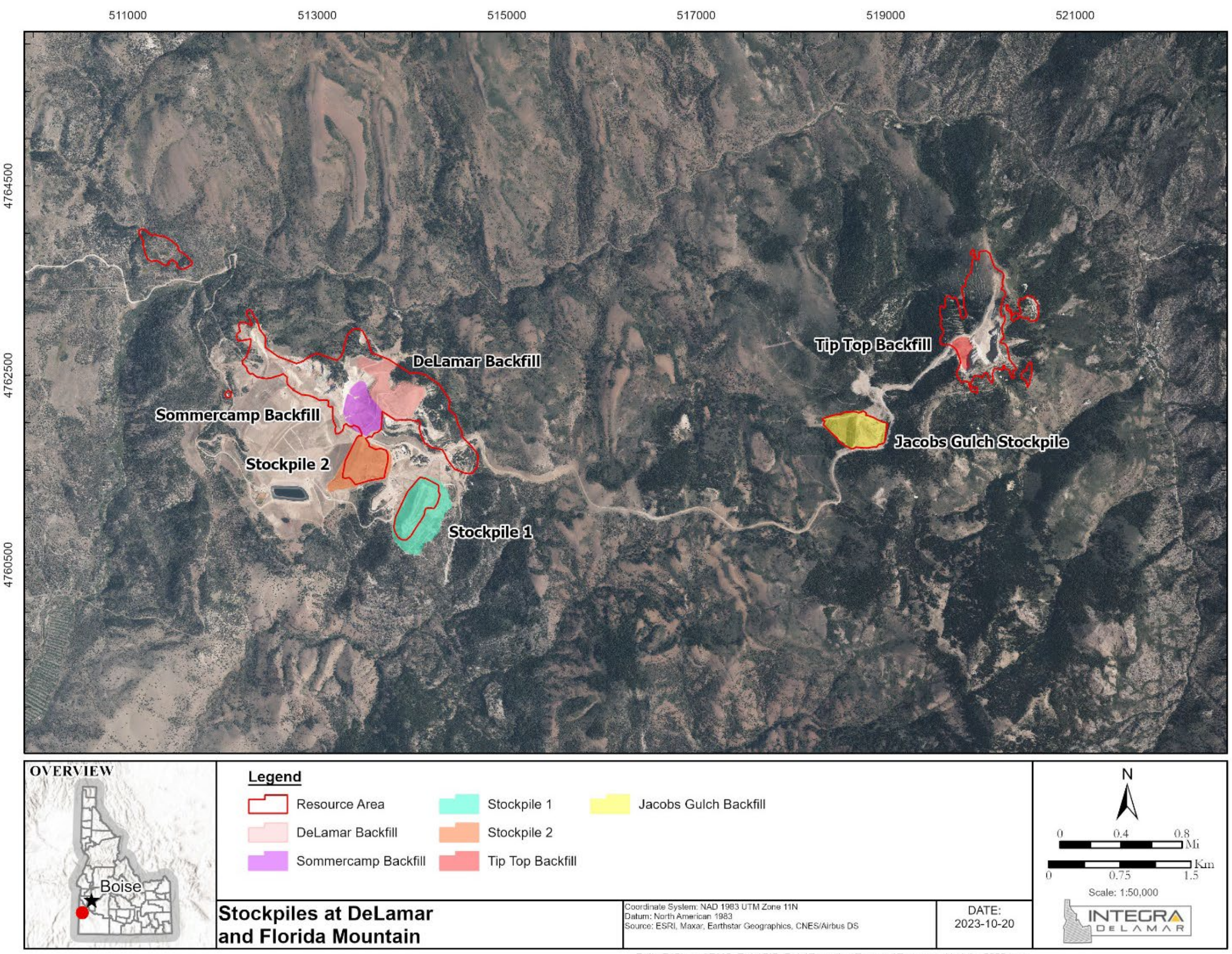
The model and estimation were checked in a similar manner as described for the DeLamar deposit estimation in Section 14.9.5.

14.11 Stockpile Gold and Silver Modeling

In addition to the estimation of in-situ (hard rock) resources, stockpile resources were also estimated. The stockpiles are comprised of materials mined, but not processed, during the historical open-pit operations, i.e., materials originally emplaced as waste dumps and pit backfill at both DeLamar and Florida Mountain. Integra completed drilling within portions of these stockpiles in 2022 and 2023 (see Section 10.4.3).

At DeLamar, the stockpiles drilled by Integra include backfill of portions of the historical North DeLamar and Sommercamp open pits and historical waste dumps #1 and #2. At Florida Mountain, the backfill within the historical TipTop open pit and the historical Jacobs Gulch waste dump were drilled (Figure 14.9).

Figure 14-9 DeLamar Project Stockpile Locations



Integra provided RESPEC with wireframe solids of each of the DeLamar project stockpiles that were drilled, as well as a summary of Integra's compilation of the original construction of each of the stockpile areas. For example, the TipTop backfill was created by end-dumping from haul trucks from the lip of the pit, with the end-dumping occurring at different positions at different times during the construction of the backfill. Integra compiled this information using historical records and interviews with ex-employees of the historical mining operations. Based on this information, Integra then created digital surfaces and solids that model the construction of each stockpile area. Mr. Gustin utilized these solids, Integra's explanatory report, and the drill-hole data in the estimation of the stockpile resources.

While Jacobs Gulch and TipTop are composed essentially entirely of oxide and mixed materials, most other stockpile areas include various, but always minor, quantities of non-oxide. The largest volume of non-oxide occurs within the North DeLamar-Sommercamp backfill. The distribution of this material is irregular, which precludes confident modeling, but it is very low grade and represents a very small percentage of the backfill volume.

While relatively higher- and lower-grade areas are present, drilled portions of the stockpiles are characterized by gold and silver grades that are quite uniform. Modeling of the gold and silver consisted of defining the limits of mineralized material within each stockpile, which also reflects the limits of the drilled areas, as well as the definition of zones of higher-than-average grade where they were found to have logical continuity in terms of the construction of the stockpile.

The outer limits of mineralized stockpile and the higher-grade zones within these limits were modeled on series of cross section for each stockpile area, resulting in an outer low-grade shell within which continuous zones of higher-grade were modeled, analogous to the mineral-domain modeling of the in-situ resources. These low- and higher-grade sectional polygons were then either sliced for rectification on long-sections or made into wireframe solids, with the long sections or solids used to code the block models, again analogous to the in-situ modeling. The higher-grade zones that were modeled were estimated independently of the remaining mass of lower-grade stockpile material to mitigate the potential overstatement of grade within the adjacent materials (grade smearing).

Table 14-17 shows the capping of assays by gold and silver grade zone for each of the stockpile areas that contribute to the project resources. The grade zones use similar codes as the in-situ mineral domains for the sake of simplicity (zone 0 represents stockpile assays lying outside of low-grade zone 100 that envelopes mineralized stockpile areas; zone 200 is assigned to the modeled higher-grade zones that lie within zone 100).

Table 14-17 Stockpile Gold and Silver Assay Caps by Zone

DeLamar Stockpile Area	Zone	g Au/t	g Ag/t	Florida Mtn Stockpile Area	Zone	g Au/t	g Ag/t
North DeLamar - SC Backfill	0	0.1	20	TipTop Backfill	0	0.35	10
	100	1.0	60		100	1.2	35
	200	2.7	140		200	2	40
DeLamar Stockpile #1	0	0.1	10	Jacobs Gulch	0	-	15
	100	1.0	65		100	1.3	60
	200	1.5	170		200	10	120
DeLamar Stockpile #2	0	0.1	10				
	100	1.0	65				
	200	1.5	170				

The capped gold and silver assays were each composited at 3.048-meter (10-foot) intervals that respect the grade-zone boundaries. Grades were estimated by inverse-distance to the second power, with nearest-neighbor and ordinary krige estimates also completed for the purposes of checking the inverse-distance estimation. Each of the five stockpile areas were estimated independently. Search restrictions were implemented during gold and silver interpolations of zone 0 and 100 for all stockpiles, with the two Florida Mountain stockpile areas also using search restrictions on the higher-grade zones (200).

14.12 DeLamar Project Mineral Resources

The DeLamar project mineral resources have been estimated to reflect potential open-pit extraction and potential processing by a variety of methods, including: crushing and heap leaching of oxide and mixed materials at DeLamar and Florida Mountain; grinding, flotation, ultra-fine regrind of concentrates, and Albion cyanide-leach processing of the reground concentrates for the non-oxide materials at DeLamar; and grinding, flotation, ultra-fine regrind of concentrates, and agitated cyanide-leaching of non-oxide materials at Florida Mountain. To meet the requirement of having reasonable prospects for eventual economic extraction by open-pit methods, pit optimizations for the DeLamar and Florida Mountain areas were run using the parameters summarized in Table 14-18 and Table 14-19 and the resulting pits were used to constrain the project resources.

Table 14-18 Resource Pit Optimization Cost Parameters

Parameter	DeLamar In Situ	Florida Mtn In Situ	N DM-SC Stockpile	DM #1 + #2 Stockpile	Jacobs Gulch Stockpile	Unit
Mining Cost	$2.00	$2.00	$1.70	$1.70	$1.70	$/tonne mined
Heap Leach						
Oxide Processing	$2.75	$2.75	$5.00	$4.25	$4.00	$/tonne processed
Mixed Processing	$3.75	$3.50	$5.00	$4.25	$4.00	$/tonne processed
Incremental Haulage	$0.20	$0.20	$0.20	$0.20	$0.20	$/tonne processed
G&A Cost	$0.40	$0.40	$0.40	$0.40	$0.40	$/tonne processed
Mill – DeLamar Area						
Non-Oxide Processing	$16.75	$9.75				$/tonne processed
Incremental Haulage	$0.20	$0.20				$/tonne processed
G&A Cost	$0.25	$0.25				$/tonne processed
Au Price	$1,800					$/oz produced
Ag Price	$21.00					$/oz produced
Au Refining Cost	$5.00					$/oz produced
Ag Refining Cost	$0.50					$/oz produced
Royalty	Table 4-2					NSR

Table 14-19 Resource Pit-Optimization Metal Recoveries

	DeLamar In Situ			Florida Mountain In Situ			DeLamar Stockpiles		Florida Mtn Stockpiles	
Process Type	Oxide	Mixed	Non-Oxide	Oxide	Mixed	Non-Oxide	N DM-SC	DM #1	DM #2	ALL
Heap Leach – Au	90%	70%	-	90%	75%	-	70%	80%	80%	90%
Heap Leach – Ag	40%	50%	-	55%	60%	-	60%	50%	55%	45%
Mill - Albion - Glen Silver - Au	-	-	78%	-	-	-	-	-	-	-
Mill - Albion - Glen Silver - Ag	-	-	78%	-	-	-	-	-	-	-
Mill - Albion – Milestone - Au	-	-	70%	-	-	-	-	-	-	-
Mill - Albion - Milestone - Ag	-	-	75%	-	-	-	-	-	-	-
Mill - Albion - Other Areas - Au	-	-	87%	-	-	-	-	-	-	-
Mill - Albion - Other Areas - Ag	-	-	87%	-	-	-	-	-	-	-
Mill - Agitated Leach - Au	-	-	-	-	-	95%	-	-	-	-
Mill - Agitated Leach - Au	-	-	-	-	-	92%	-	-	-	-

The pit shells created using the 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 0.17 g/t to all in-situ model blocks lying within the optimized pits that are coded as oxide or mixed, a 0.1 g/t gold-equivalent cutoff to all stockpile materials, a 0.3 g/t gold-equivalent cutoff to in-situ blocks coded as non-oxide at DeLamar, and a 0.2 g/t cutoff to in-situ blocks coded as non-oxide at Florida Mountain. The in-situ oxide and mixed gold and silver minimum grade of 0.17 g/t was used as an override on the pit optimizations of in situ material, so that all blocks below the cutoff were treated as 'waste' during the optimization run. The resource cutoff applied to non-oxide materials at Florida Mountain is lower than that at DeLamar, which is due to the lower processing costs and higher recoveries attributed to these materials at Florida Mountain.

Gold equivalency is a function of metal recoveries (Table 14-18) and metal recoveries (Table 14-19), with the recoveries varying by deposit and oxidation state. 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 a Florida Mountain model block coded as mixed is as follows:

$$\text{g AuEq/t} = \text{g Au/t} + (\text{g Ag/t} \div ((1{,}800 \times 0.75) \div (21 \times 0.60)))$$

where "g Au/t" and "g Ag/t" are the estimated gold and silver grades, respectively, and the other variables are the metal prices and recoveries. The gold-equivalent grades are calculated for each block for the sole purpose of applying the resource cutoffs defined above to the appropriate materials within the optimized pits. As an example, Table 14-20 lists the gold-equivalent factors applied to the in-situ resources.

Table 14-20 Gold-Equivalency Factors Applied to Silver Grades

DeLamar			Florida Mountain		
Oxide	Mixed	Non-Oxide	Oxide	Mixed	Non-Oxide
161	171	85	118	132	88

The total DeLamar project mineral resources, which include the in-situ and stockpile resources for both the DeLamar and Florida Mountain areas, are summarized in Table 14-21.

Table 14-21 Total DeLamar Project Gold and Silver Resources

Type	Class	Tonnes	Au g/t	Au oz	Ag g/t	Ag oz
Oxide	Measured	6,313,000	0.36	74,000	16.9	3,427,000
	Indicated	42,346,000	0.35	471,000	13.4	18,291,000
	Inferred	11,132,000	0.28	99,000	7.8	2,795,000
	Meas + Ind	48,659,000	0.35	545,000	13.9	21,718,000
Mixed	Measured	10,043,000	0.42	136,000	21.8	7,032,000
	Indicated	60,136,000	0.35	672,000	15.0	29,010,000
	Inferred	8,533,000	0.27	74,000	8.4	2,302,000
	Meas + Ind	70,179,000	0.37	808,000	16.5	36,042,000
NonOxide	Measured	21,056,000	0.51	345,000	32.8	22,198,000
	Indicated	65,486,000	0.45	943,000	22.2	46,640,000
	Inferred	18,561,000	0.38	229,000	14.0	8,371,000
	Meas + Ind	86,542,000	0.46	1,288,000	24.7	68,838,000
Stockpiles	Measured	-	-	-	-	-
	Indicated	42,455,000	0.22	296,000	11.8	16,149,000
	Inferred	4,877,000	0.17	26,000	9.8	1,535,000
	Meas + Ind	42,455,000	0.22	296,000	11.8	16,149,000
Total Resources	Measured	37,412,000	0.46	554,000	27.2	32,657,000
	Indicated	210,424,000	0.35	2,381,000	16.3	110,091,000
	Inferred	43,101,000	0.31	428,000	10.8	15,002,000
	Meas + Ind	247,836,000	0.37	2,935,000	18.1	142,748,000

1. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*
2. *Michael M. Gustin, C.P.G. and Principal Consultant for RESPEC, is a Qualified Person as defined in NI 43-101, and is responsible for reporting mineral resources in this technical report. Mr. Gustin is independent of Integra.*
3. *In-Situ Oxide and Mixed and all Stockpile mineral resources are reported at a 0.17 and 0.1 g AuEq/t cut-off, respectively, in consideration of potential open-pit mining and heap-leach processing.*
4. *Non-Oxide mineral resources are reported at a 0.3 g AuEq/t cut-off at DeLamar and 0.2 g AuEq/t at Florida Mountain in consideration of potential open pit mining and grinding, flotation, ultra-fine regrind of concentrates, and either Albion or agitated cyanide-leaching of the reground concentrates.*
5. *The mineral resources are constrained by pit optimizations.*
6. *Gold equivalent grades were calculated using the metal prices and recoveries presented in Table 14.18 and Table 14.19.*
7. *Rounding as required by reporting guidelines may result in apparent discrepancies between tonnes, grades, and contained metal content.*
8. *The effective date of the mineral resources is August 25, 2023.*
9. *The estimate of mineral resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

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

The current project mineral resources are inclusive of the mineral reserves discussed in Section 15. The mineral reserve statement included herein has an effective date of January 24, 2022 and is unaffected by the mineral resource update included herein.

The DeLamar project resources are classified according to the criteria presented in Table 14-22.

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 meet the resource constraints
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

The Measured and Indicated classification constraints for the Florida Mountain area are more restrictive than those for the DeLamar area. This is due to the differing styles of higher-grade mineralization in each of the deposits. At Florida Mountain, higher-grade mineralization occurs as irregular, large-scale stockwork zones of limited continuity that require a higher density of drilling to properly define than is the case with the more regular and continuous higher-grade mineralization at the DeLamar deposit.

Although the authors are not experts 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.

The gold and silver resources for the DeLamar and Florida Mountain areas are reported separately in Table 14-23 and Table 14-24, respectively.

Table 14-23 Gold and Silver Resources of the DeLamar Area

Type	Class	Tonnes	Au g/t	Au oz	Ag g/t	Ag oz
Oxide	Measured	3,178,000	0.34	35,000	18.6	1,904,000
	Indicated	23,756,000	0.32	246,000	17.0	12,994,000
	Inferred	4,992,000	0.28	46,000	11.1	1,781,000
	Meas + Ind	26,934,000	0.32	281,000	17.2	14,898,000
Mixed	Measured	4,126,000	0.40	53,000	36.2	4,797,000
	Indicated	27,063,000	0.31	274,000	21.6	18,766,000
	Inferred	2,932,000	0.27	26,000	12.9	1,220,000
	Meas + Ind	31,189,000	0.35	327,000	24.7	23,563,000
NonOxide	Measured	16,541,000	0.54	288,000	38.1	20,249,000
	Indicated	48,608,000	0.45	710,000	26.4	41,292,000
	Inferred	11,797,000	0.41	157,000	17.0	6,456,000
	Meas + Ind	65,149,000	0.48	998,000	29.4	61,541,000
Stockpiles	Measured	-	-	-	-	-
	Indicated	30,623,000	0.19	188,000	13.7	13,506,000
	Inferred	4,019,000	0.17	21,000	11.2	1,449,000
	Meas + Ind	30,623,000	0.19	188,000	13.7	13,506,000
Total Resources	Measured	23,845,000	0.49	375,000	35.2	26,950,000
	Indicated	130,051,000	0.34	1,417,000	20.7	86,558,000
	Inferred	23,739,000	0.33	250,000	14.3	10,905,000
	Meas + Ind	153,896,000	0.37	1,792,000	23.2	113,508,000

Notes:

1. *Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.*
2. *Michael M. Gustin, C.P.G. and Principal Consultant for RESPEC, is a Qualified Person as defined in NI 43-101, and is responsible for reporting mineral resources in this technical report. Mr. Gustin is independent of Integra.*
3. *In-Situ Oxide and Mixed and all Stockpile mineral resources are reported at a 0.17 and 0.1 g AuEq/t cut-off, respectively, in consideration of potential open-pit mining and heap-leach processing.*
4. *Non-Oxide mineral resources are reported at a 0.3 g AuEq/t cut-off at DeLamar and 0.2 g AuEq/t at Florida Mountain in consideration of potential open pit mining and grinding, flotation, ultra-fine regrind of concentrates, and either Albion or agitated cyanide-leaching of the reground concentrates.*
5. *The mineral resources are constrained by pit optimizations.*
6. *Gold equivalent grades were calculated using the metal prices and recoveries presented in Table 14.18 and Table 14.19.*
7. *Rounding as required by reporting guidelines may result in apparent discrepancies between tonnes, grades, and contained metal content.*
8. *The effective date of the mineral resources is August 25, 2023.*
9. *The estimate of mineral resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*

Table 14-24 Gold and Silver Resources of the Florida Mountain Area

Type	Classification	Tonnes	Au g/t	Au oz	Ag g/t	Ag oz
Oxide	Measured	3,135,000	0.39	39,000	15.1	1,523,000
	Indicated	18,590,000	0.38	225,000	8.9	5,297,000
	Inferred	6,140,000	0.27	53,000	5.1	1,014,000
	Meas + Ind	21,725,000	0.38	264,000	9.8	6,820,000
Mixed	Measured	5,917,000	0.44	83,000	11.8	2,235,000
	Indicated	33,073,000	0.38	398,000	9.6	10,244,000
	Inferred	5,601,000	0.27	48,000	6.0	1,082,000
	Meas + Ind	38,990,000	0.38	481,000	10.0	12,479,000
NonOxide	Measured	4,515,000	0.39	57,000	13.4	1,949,000
	Indicated	16,878,000	0.43	233,000	9.9	5,348,000
	Inferred	6,764,000	0.33	72,000	8.8	1,915,000
	Meas + Ind	21,393,000	0.42	290,000	10.6	7,297,000
Stockpiles	Measured	-	-	-	-	-
	Indicated	11,832,000	0.28	108,000	6.9	2,643,000
	Inferred	858,000	0.19	5,000	3.1	86,000
	Meas + Ind	11,832,000	0.28	108,000	6.9	2,643,000
Total Resources	Measured	13,567,000	0.41	179,000	13.1	5,707,000
	Indicated	80,373,000	0.37	964,000	9.1	23,533,000
	Inferred	19,362,000	0.29	178,000	6.6	4,097,000
	Meas + Ind	93,940,000	0.38	1,143,000	9.7	29,240,000

Notes:

1. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*
2. *Michael M. Gustin, C.P.G. and Principal Consultant for RESPEC, is a Qualified Person as defined in NI 43-101, and is responsible for reporting mineral resources in this technical report. Mr. Gustin is independent of Integra.*
3. *In-Situ Oxide and Mixed and all Stockpile mineral resources are reported at a 0.17 and 0.1 g AuEq/t cut-off, respectively, in consideration of potential open-pit mining and heap-leach processing.*
4. *Non-Oxide mineral resources are reported at a 0.3 g AuEq/t cut-off at DeLamar and 0.2 g AuEq/t at Florida Mountain in consideration of potential open pit mining and grinding, flotation, ultra-fine regrind of concentrates, and either Albion or agitated cyanide-leaching of the reground concentrates.*
5. *The mineral resources are constrained by pit optimizations.*
6. *Gold equivalent grades were calculated using the metal prices and recoveries presented in Table 14.18 and Table 14.19.*
7. *Rounding as required by reporting guidelines may result in apparent discrepancies between tonnes, grades, and contained metal content.*
8. *The effective date of the mineral resources is August 25, 2023.*
9. *The estimate of mineral resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*

The oxide and mixed heap-leach portion of the total project Measured and Indicated resources contain 1,649,000 ounces of gold and 73,909,000 ounces of silver. Inferred project heap-leach resources include 199,000 ounces of gold and 6,632,000 ounces of silver.

Figure 14.10 through Figure 14.13 are representative cross-sections showing the estimated block-model gold and silver grades for the DeLamar area. Figure 14.14 through Figure 14.17 are representative cross-sections showing the estimated block-model gold and silver grades for the Florida Mountain area. These figures correspond to the in situ mineral domain cross-sections presented in Figure 14.1 through Figure 14.4 and Figure 14.5 through Figure 14.8 for DeLamar and Florida Mountain, respectively.

Figure 14-10 Cross Section 2190 NW Showing Sullivan Gulch Block-Model Gold Grades

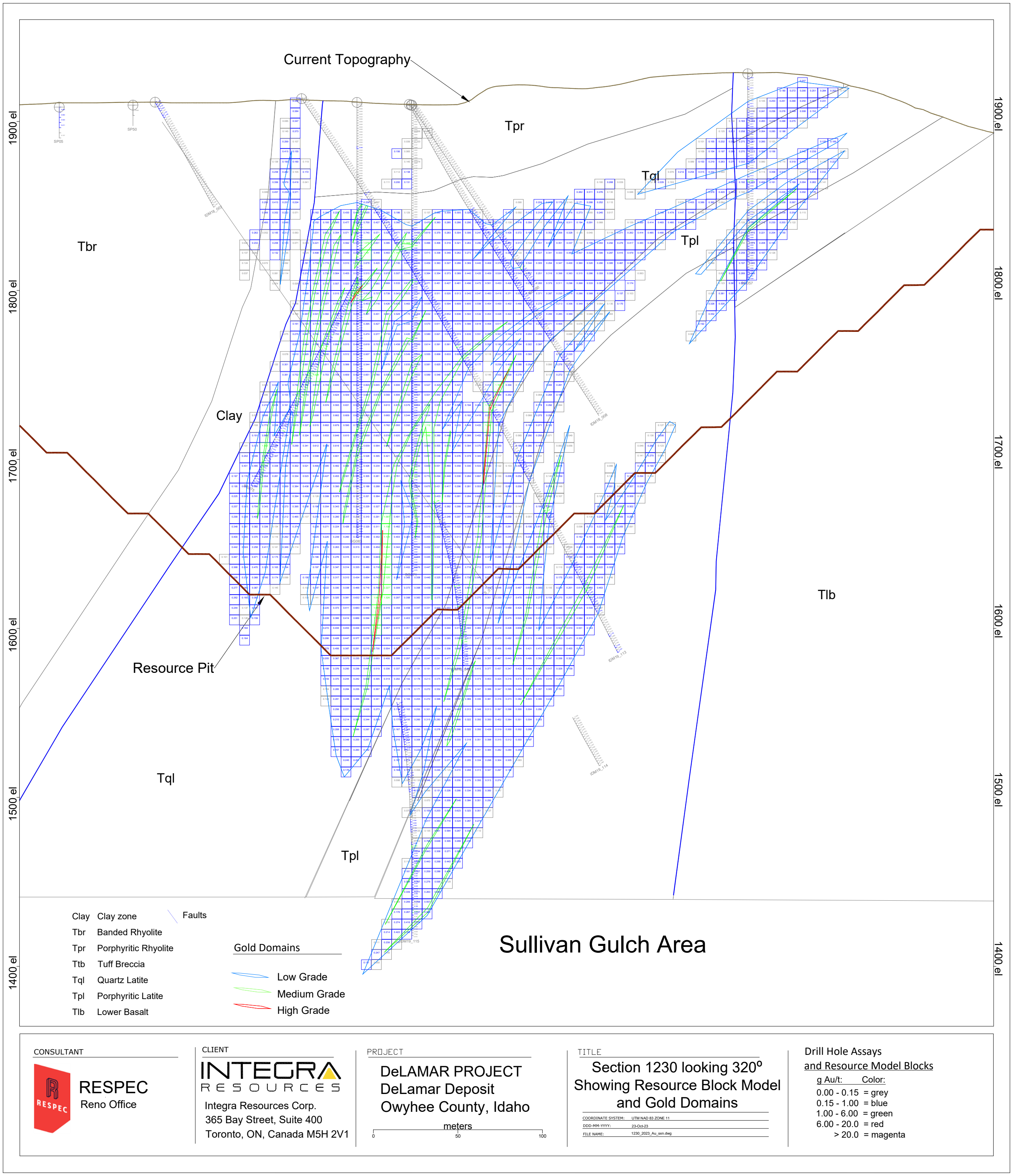


Figure 14-11 Cross Section 2190 NW Showing Sullivan Gulch Block-Model Silver Grades

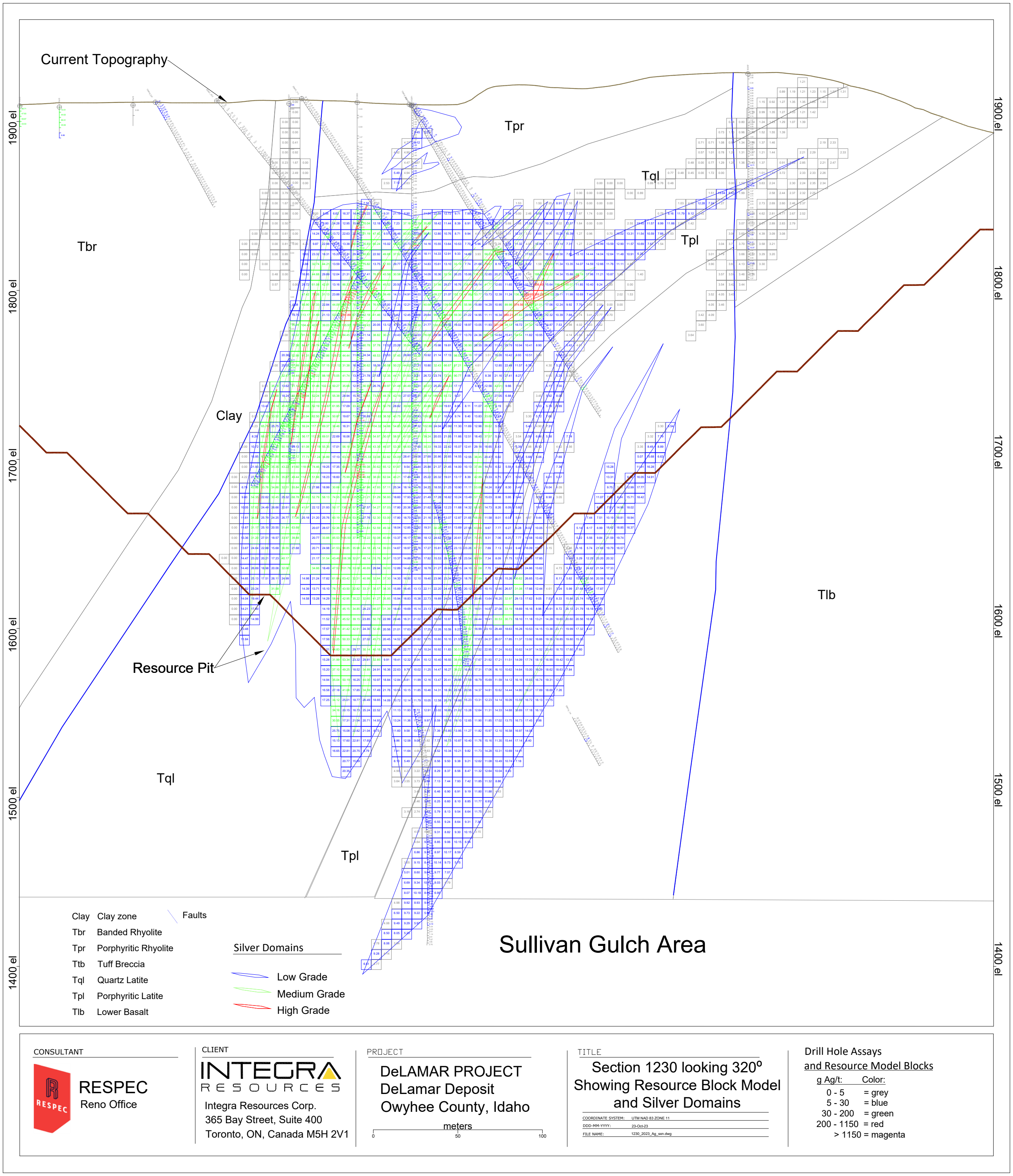


Figure 14-12 Cross Section 2790 NW Showing Glen Silver and South Wahl Block-Model Gold Grades

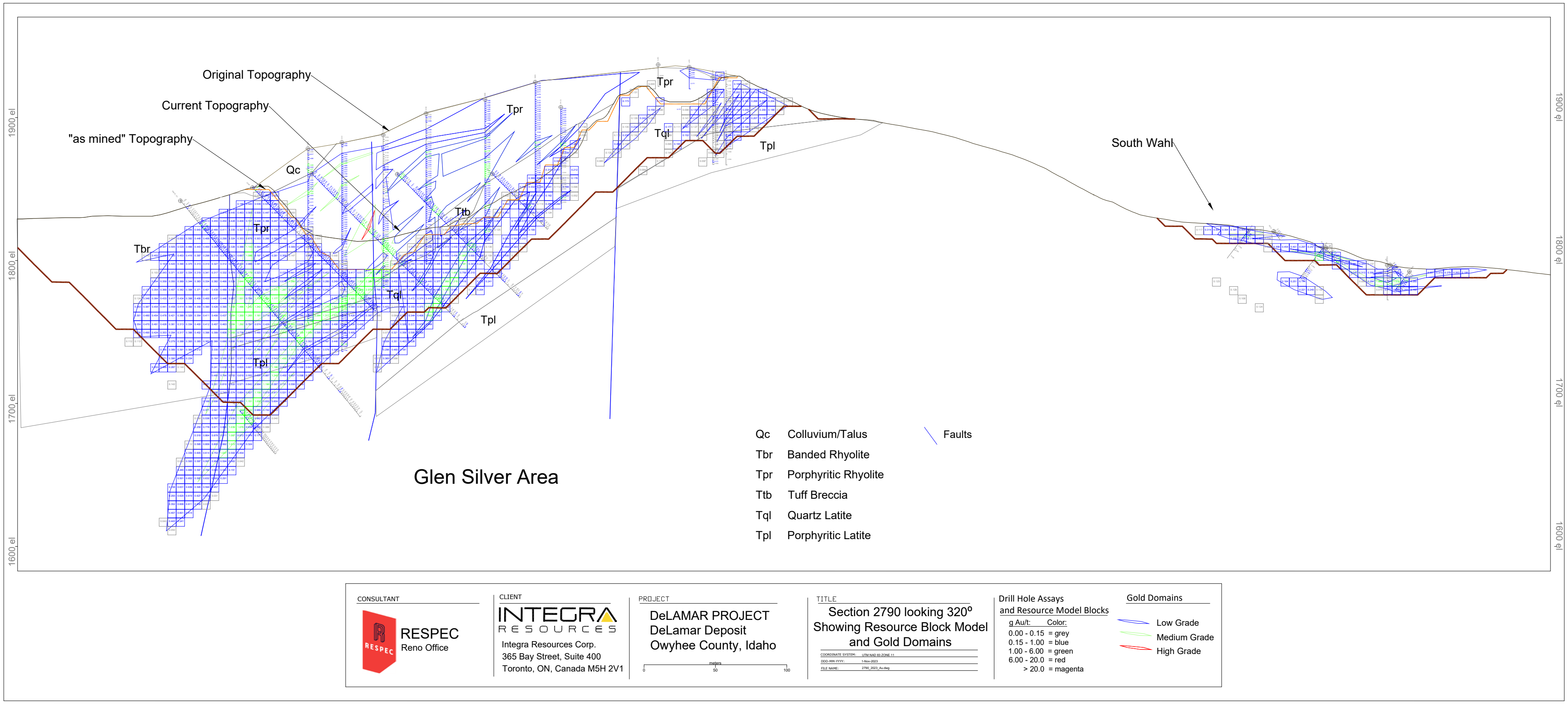


Figure 14-13 Cross Section 2790 NW Showing Glen Silver and South Wahl Block-Model Silver Grades

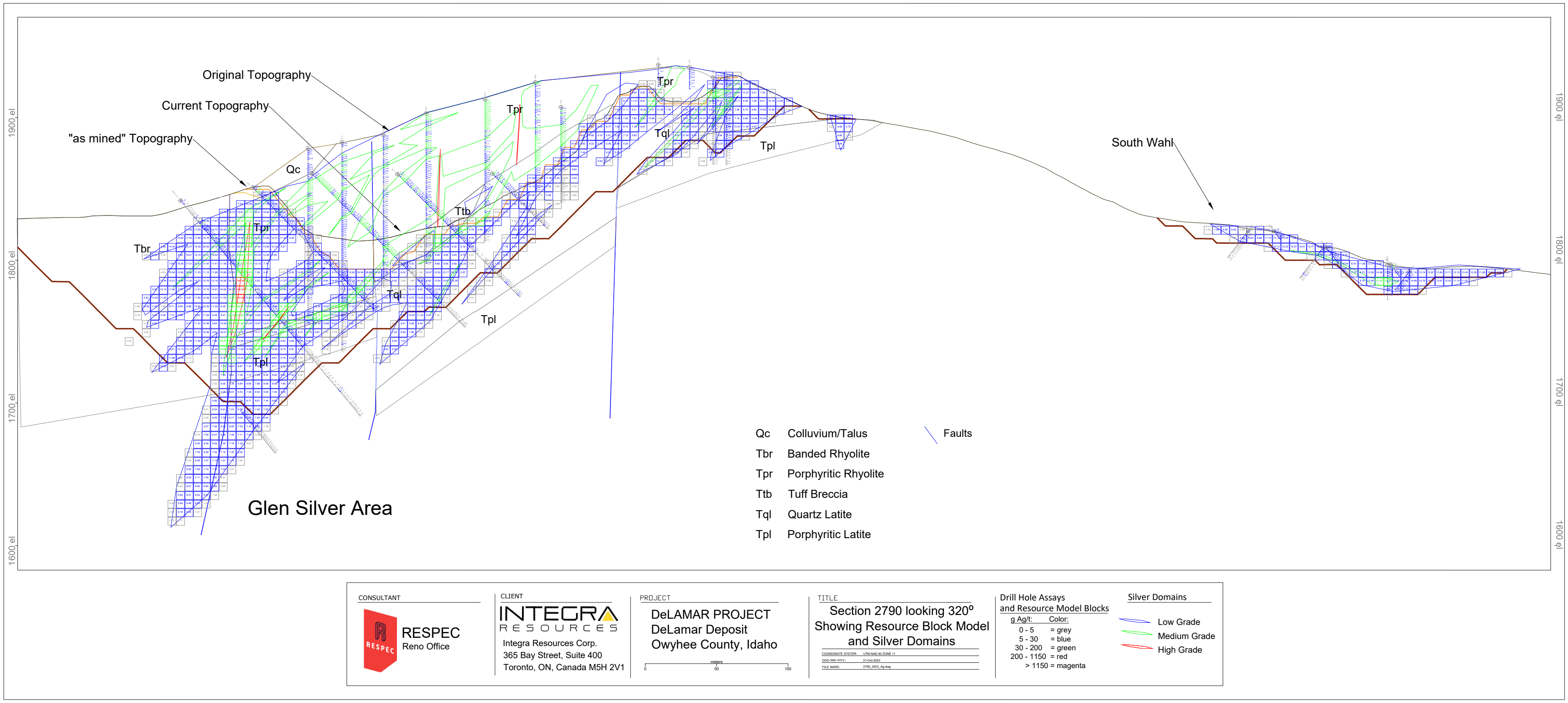


Figure 14-14 Cross Section 2830 N Showing Florida Mountain Block-Model Gold Grades

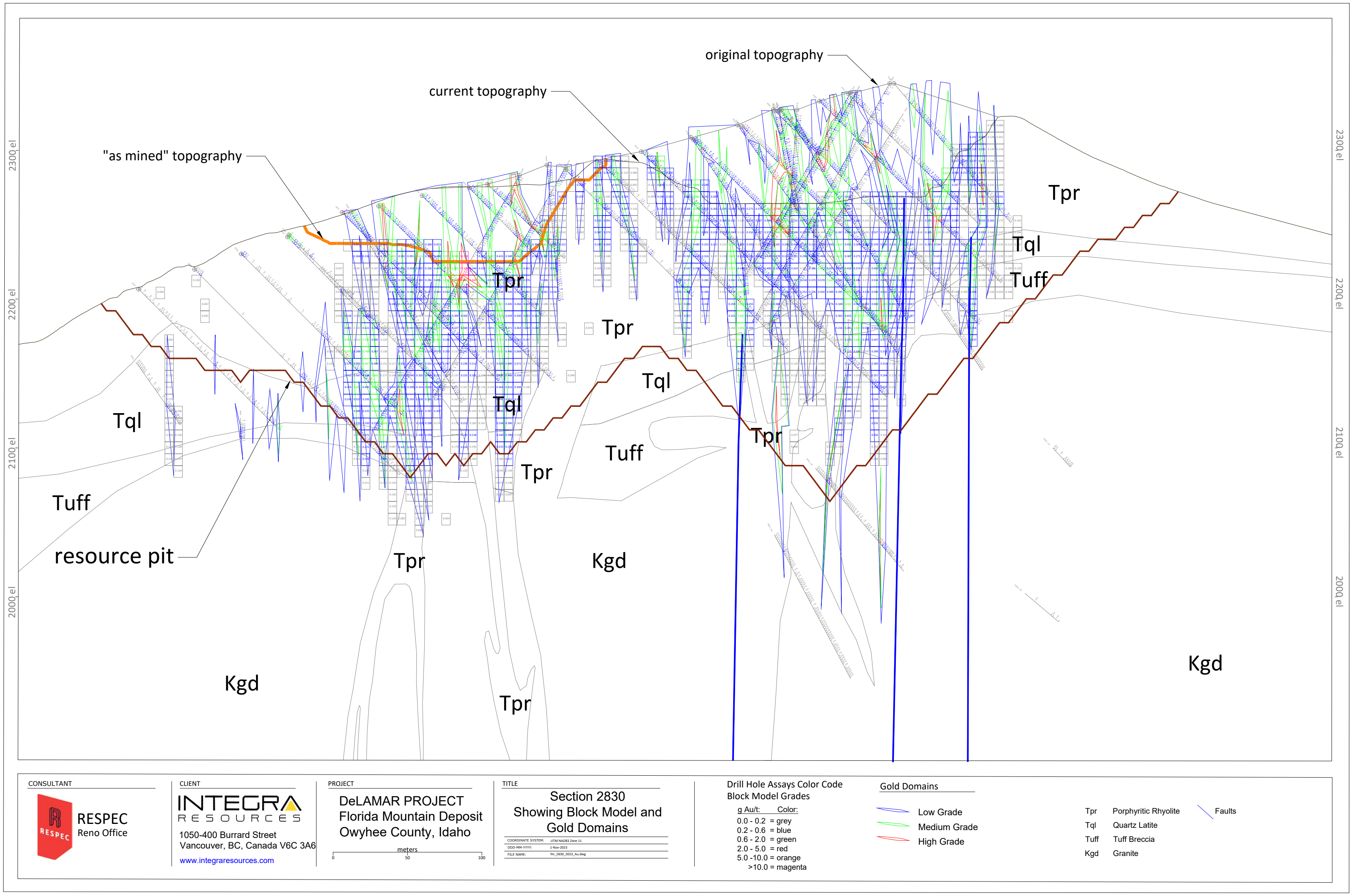


Figure 14-15 Cross Section 2830 N Showing Florida Mountain Block-Model Silver Grades

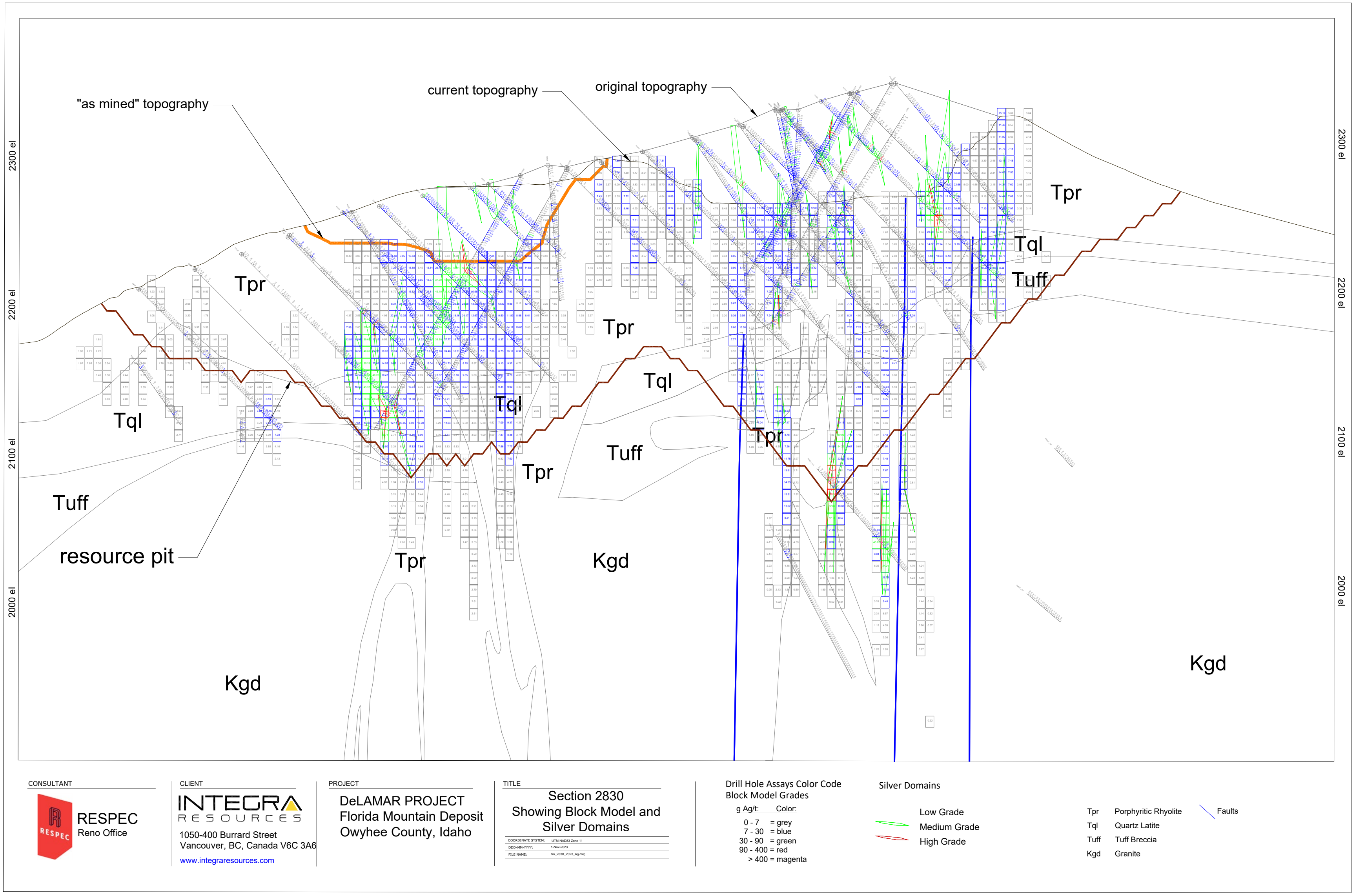


Figure 14-16 Cross Section 3280 N Showing Florida Mountain and Block-Model Gold Grades

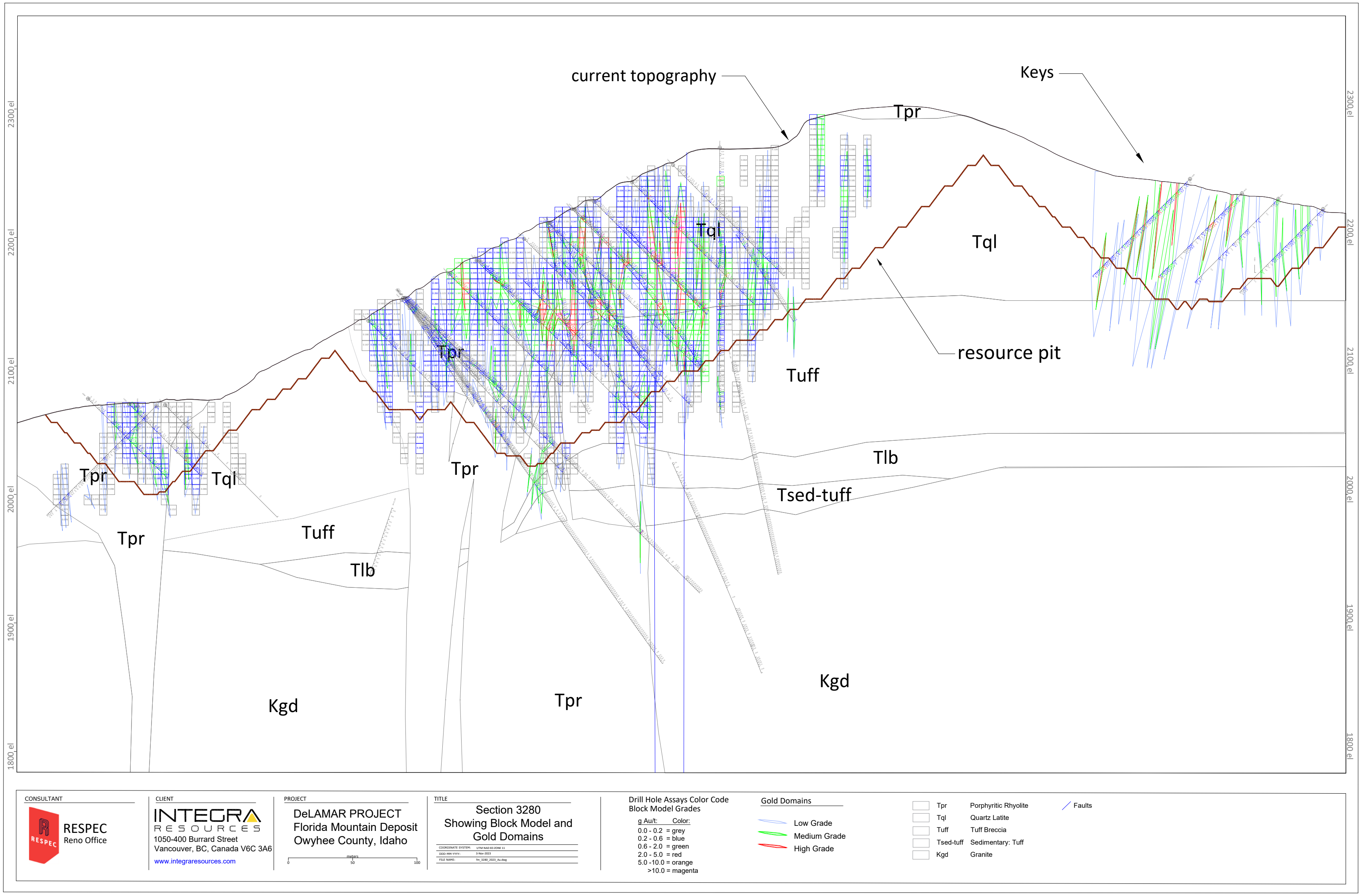
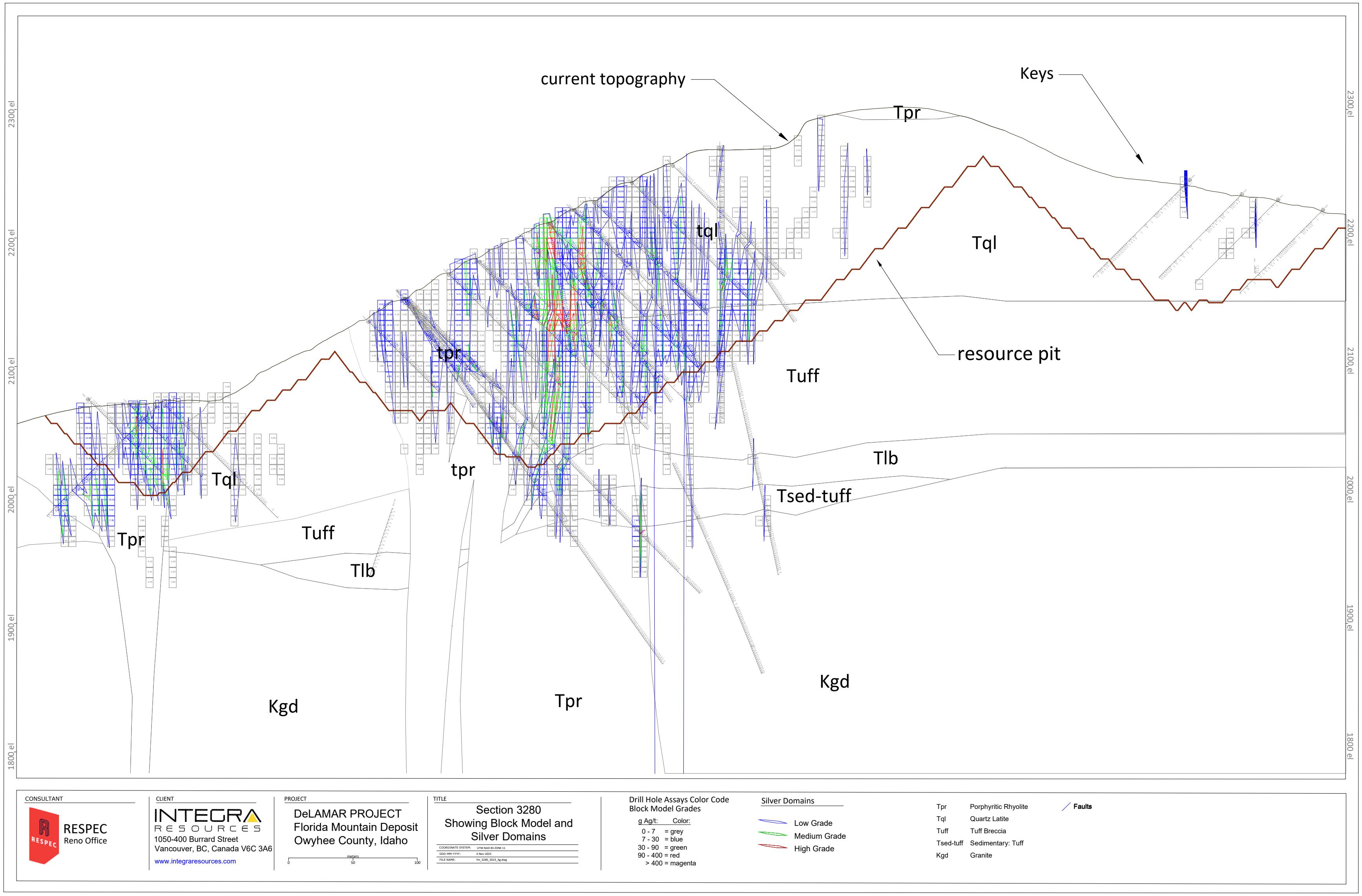


Figure 14-17 Cross Section 3280 N Showing Florida Mountain and the Keys Block-Model Silver Grades



14.13 Discussion of Resource Modeling

The current resources described in this report differ from those reported in 2022 primarily due to the first-time estimation of stockpile resources at both the DeLamar and Florida Mountain areas and the addition of the Keys resources, which lie to the west of the Florida Mountain resources proper, as well as some smaller areas beyond the limits of the main Florida Mountain resource area that would need to expand with more drilling in order to have a chance of conversion into reserves.

Consistent with assaying methods at the time, 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 Au/ton), and the quality of historical assays that had detection limits below 0.17 g Au/t are uncertain. Current economic parameters for heap-leach processing can lead to mining cutoffs that are equal to or less than this grade. Due to the uncertainty with respect to the historical assays at low grades, a 0.17 g Au/t override was applied to in-situ materials for the resource-pit optimizations, such that some low-grade in-situ blocks that might otherwise be designated for processing based on the input parameters are overridden as 'waste' in the pit optimizations. The 0.17 g Au/t cutoff is also the resource cutoff grade applied to in-pit, in-situ, oxide and mixed blocks. Any potential future mining operation that may consider the use of mining cutoff grades below this cutoff grade should consider the uncertainties inherent in the historical assay database.

The drilling that forms the basis of the resource estimations was done primarily by RC and, to a lesser extent, 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 resource estimation of the DeLamar area. However, potentially contaminated samples may remain in the data used in the estimations, although the possible inclusion of such samples is not considered to be a material issue at DeLamar. No down-hole contamination was recognized within the Florida Mountain resources, which lie above the water table.

The late-1800s to early-1900s underground stopes in the DeLamar area were almost entirely mined out by 1977 through 1998 historical open-pit mining operations. 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 RESPEC, extend into the Florida Mountain resources. A total of approximately 200,000 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 resources, it is not uncommon for drill holes to have markedly different grades than adjacent holes, which is not surprising given that the mineralization is in the form of a mega-stockwork. Mr. Gustin believes the explicit modeling of the gold and silver domains, combined with the tight drill spacing at Florida Mountain, where a high percentage of resource blocks lie within an average distance of 20 meters (65 feet) from two drill holes, has led to properly representing this inherent geologic variability.

There are logged areas of "garbage" and "oil" within certain areas of the DeLamar stockpiles #1 and #2 that were intentionally buried within the waste dumps during the historical mining operations. While almost all these areas are very low grade and, even if of sufficient grade, excluded from the resources, some isolated logged trash 'intercepts' remain in the resources.

15.0 MINERAL RESERVE ESTIMATES

Mr. Dyer, P.E., the responsible qualified person for the mineral reserve estimate in the PFS technical report and included in this technical report, reviewed the updated mineral resource model and determined that the updated mineral resource model does not materially change the mineral reserve statement. In particular, there have been no material changes to the portions of the mineral resource base that support the mineral reserves. This conclusion is based on: (i) the mineral reserve statement did not include stockpile materials that have been added to the current mineral resources; (ii) exclusive of the stockpile materials, the current DeLamar area in situ Measured and Indicated mineral resources potentially amenable to heap-leach and milling are not materially different from previous mineral resource statement; (iii) the increases in the current Florida Mountain in situ Measured and Indicated mineral resources potentially amenable to heap leach and milling are due to the first-time inclusion of mineral resources from the Keys area; (iv) the Keys mineral resources currently lack sufficient metallurgical testing, geotechnical work and other engineering studies that would be required to support mineral reserves; and (v) Integra has no intention of developing mineral reserves in the Keys area currently or in the foreseeable future. As noted above, the Keys area was included in the current mineral resources at the suggestion of Mr. Gustin, as the Keys mineralization has reasonable prospects for eventual economic extraction, even as Integra currently has no plans to do so.

Excluding the Keys area from the Florida Mountain resources, potential heap-leach Measured and Indicated mineral resources have increased by 3% as compared to the previous mineral resource estimate, while the current Measured and Indicated mineral resources potentially amenable to mill-related processing increased by 2%. At DeLamar, the current Measured and Indicated mineral resources increased by 4% and decreased by 3% for the heap leach and mill scenarios, respectively. These changes in the DeLamar project mineral resources are well within the limits of accuracy inherent in the estimation of resources. Accordingly, the results of the PFS and the mineral reserve statement have been reproduced in this technical report and remain unaffected by the updated mineral resource. The PFS and mineral reserve statement have an effective date of January 24, 2022.

15.1 Geotechnical Parameters

RESPEC conducted a geotechnical investigation and engineering analysis to develop recommendations for bench-face angles (“BFAs”), inter-ramp angles (“IRAs”), overall slope angles (“OSA”s), and catch-bench widths for the recommended planned pit slopes. This investigation and engineering analysis was conducted under the supervision of Mr. Jay Nopola of RESPEC’s Rapid City, South Dakota office. The results were reported in Raffaldi et al. (2021) and were based on the following information:

- Historical geotechnical data;
- Laboratory rock-mechanics tests;
- Visual observations of the performance on existing highwalls;
- Verbal and written communications with current and former site employees;
- Estimates of rock mass properties from available rock core and core photographs;
- Geological maps and three-dimensional geological models;

- High-resolution unmanned aerial vehicle ("UAV") point-cloud data; and
- The 2019 Preliminary Economic Assessment ("PEA") of Gustin et al. (2019) and associated pit designs.

RESPEC's PFS-level recommendations for pit slope design varies from 40 to 45° at DeLamar and from 30 to 50° for Florida Mountain are summarized in Table 15.3 and Table 15.4. Recommended layback angles for slopes in soil materials throughout the site are 1.75:1 for Qcl (≤ 60 m) and 2:1 for other material types or higher heights.

15.2 Mineral Reserve Statement

Mr. Dyer has used Measured and Indicated in situ mineral resources as the basis to define mineral reserves for both the DeLamar and Florida Mountain deposits. Mineral reserve definition was done by first identifying ultimate pit limits using economic parameters and pit optimization techniques. The resulting optimized pit shells were then used for guidance in pit design to allow access for equipment and personnel. Mr. Dyer then considered mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, and governmental factors for defining the estimated mineral reserves.

The economic parameters and cutoff grades used in the estimation of the mineral reserves are shown in Table 15-1. The overall leaching process rate is planned to be 35,000 tonnes (38,581 tons) per day or 12,600,000 tonnes (13,889,123 tons) per year for both Florida Mountain and DeLamar oxide and mixed material. DeLamar leach processing will also include agglomeration. Initially only the oxide and mixed material will be processed, then starting in year 3, non-oxide will be processed through a plant constructed to operate at a rate of 6,000 tonnes (6,614 tons) per day or 2,160,000 tonnes (2,380,992 tons) per year.

The cutoff grades applied reflect the cost to process material along with G&A and incremental haulage costs. Note that royalties are built into the block values and are considered in determining whether to process the material. While the DeLamar non-oxide breakeven cutoff grade would be $11.44/t according to the applicable costs, a cutoff of $15.00 was assigned to enhance the project's economic performance.

Table 15-1 DeLamar and Florida Mountain Economic Parameters

	DeLamar			Florida Mnt			
	Oxide	Mixed	Non-Oxide	Oxide	Mixed	Non-Oxide	Units
Mining Cost	$ 2.00	$ 2.00	$ 2.00	$ 2.00	$ 2.00	$ 2.00	$/t Mined
Incremental Ore Haulage	$ 0.20	$ 0.20	$ 0.20	$ 0.20	$ 0.20	$ 0.20	$/t Processed
Process Cost	$ 3.00	$ 4.00	$ 11.02	$ 2.75	$ 3.50	$ 9.00	$/t Processed
G&A	$ 0.44	$ 0.44	$ 0.22	$ 0.45	$ 0.45	$ 0.25	$/t Processed
GMV Breakeven COG	$ 3.64	$ 4.64	$ 11.44	$ 3.40	$ 4.15	$ 9.45	$/t Processed
GMV COG Used	$ 3.65	$ 4.65	$ 15.00	$ 3.55	$ 4.20	$ 10.35	$/t Processed
Final Process Costs	$ 4.27	$ 4.29	$ 11.91	$ 2.98	$ 3.67	$ 10.60	$/t Processed

GMV = gross metal value; COG = cutoff grade.

Total Proven and Probable reserves for the DeLamar project from all pit phases are 123,483,000 tonnes at an average grade of 0.45 g Au/t and 23.27 g Ag/t, for 1,787,000 ounces of gold and 92,403,000 ounces of silver (Table 15-2). The mineral reserves point of reference is the point where material is fed into the crusher.

Table 15-2 Total Proven and Probable Reserves, DeLamar and Florida Mountain

	Classification	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	Block Value
Oxide	Proven	3,295	0.39	41	17.39	1,842	19.34
	Probable	31,486	0.37	375	15.24	15,426	17.93
	P&P	34,782	0.37	416	15.44	17,268	$ 18.06
Mixed	Proven	7,741	0.49	122	25.75	6,409	23.72
	Probable	49,718	0.40	637	17.29	27,632	18.29
	P&P	57,459	0.41	759	18.43	34,042	$ 19.02
Non-oxide	Proven	7,321	0.65	153	53.15	12,511	39.33
	Probable	23,921	0.60	459	37.16	28,582	33.81
	P&P	31,243	0.61	612	40.91	41,093	$ 35.11
Total	Proven	18,358	0.54	316	35.18	20,763	$ 29.16
	Probable	105,126	0.44	1,471	21.20	71,640	$ 21.71
	P&P	123,483	0.45	1,787	23.27	92,403	$ 22.82

Notes:

1. *All estimates of Mineral Reserves have been prepared in accordance with NI 43-101 and are included within the current Measured and Indicated mineral resources.*
2. *Thomas L. Dyer, P.E. for RESPEC, a division of RESPEC, in Reno, Nevada, is a Qualified Person as defined in NI 43-101, and is responsible for reporting Proven and Probable mineral reserves for the DeLamar Project. Mr. Dyer is independent of Integra.*
3. *Mineral reserves are based on prices of $1,650 per ounce Au and $21.00 per ounce Ag. The reserves were defined based on pit designs that were created to follow optimized pit shells created in Whittle. Pit designs followed pit slope recommendations provided by RESPEC.*
4. *Reserves are reported using block value cutoff grades representing the cost of processing:*
5. *Florida Mountain oxide leach cutoff grade value of $3.55/t.*
6. *Florida Mountain mixed leach cutoff grade value of $4.20/t.*
7. *Florida Mountain non-oxide mill cutoff grade value of $10.35/t.*
8. *DeLamar oxide leach cutoff grade value of $3.65/t*
9. *DeLamar mixed leach cutoff grade value of $4.65/t.*
10. *DeLamar non-oxide mill cutoff grade value of $15.00/t.*
11. *The mineral reserves point of reference is the point where is material is fed into the crusher.*
12. *The effective date of the mineral reserves estimate is January 24, 2022.*
13. *All ounces reported herein represent troy ounces, "g Au/t" represents grams per gold tonne and "g Ag/t" represents grams per silver tonne.*
14. *Columns may not sum due to rounding.*
15. *The estimate of mineral reserves may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*
16. *Energy prices of US$2.50 per gallon of diesel and $0.065 per kWh were used.*

16.0 MINING METHODS

The PFS reproduced and presented in this report considers open-pit mining of the DeLamar and Florida Mountain gold-silver deposits. Mining would utilize 23-cubic meter (30-cubic yard) hydraulic shovels along with 13-cubic meter (16.7-cubic yard) loaders to load 136-tonne capacity haul trucks. The haul trucks would haul waste and ore out of the pit and to dumping locations. Due to the length of ore hauls, the ore will be stockpiled near the pits followed by loading into a Railveyor system which will convey the ore into a crusher. The Railveyor system will be supplemented with haul trucks on an as needed basis.

Waste material will be stored in waste-rock storage facilities ("WRSFs") located near each of the Florida Mountain and DeLamar deposits, as well as backfilled into pits where available. The exception is the Milestone pit, from which waste material will be fully utilized for construction material for the tailing storage facility ("TSF").

This section describes the locations and designs for WRSFs and backfill designs, mine production schedule, process material delivery schedule, stockpiling schedule, mine equipment requirements, and personnel requirements for the PFS.

16.1 WRSFs and Backfill Designs

WRSFs, along with backfill areas, have been designed for the PFS to contain the waste material mined from the different pit phases. Volumes of waste material mined from the Florida Mountain and DeLamar pits are shown in Table 16.1. The volumes of waste material are calculated using the SG value in the resource model along with a 1.3 swell factor. The swell factor is intended to represent the swelling of material as it is blasted and loaded into trucks along with recompacting the material as it is placed in the WRSFs and backfill areas. The WRSFs are shown in the GA drawing in Section 18.1.

Table 16-1 Waste Rock Containment Requirements (With Swell)

	Waste Volumes (K Cu M) w/Swell				
	Oxide	Mixed	Non-Oxide	Fill	Total
FlMnt_Ph_1	2,624	2,163	516	6	5,309
FlMnt_Ph_2	10,727	11,300	8,414	25	30,467
FlMnt_Ph_3	4,170	7,316	6,946	1,364	19,795
Total FlMnt	**17,521**	**20,779**	**15,876**	**1,395**	**55,571**
Del_Ph_1	3,504	1,954	2,510	1,257	9,225
Del_Ph_2	2,901	1,213	2,316	4,402	10,832
Del_Ph_3	6,690	8,284	8,459	3,048	26,481
Del_Ph_4	15,777	9,837	29,930	-	55,544
Del_Ph_5	2,584	443	769	-	3,796
Total Del	**31,456**	**21,732**	**43,983**	**8,707**	**105,878**
Total Project	**48,977**	**42,511**	**59,859**	**10,102**	**161,449**

Note: FlMtn = Florida Mountain; Del = DeLamar.
Swell factor of 1.3 was used for containment requirements

Waste rock storage capacities are shown in Table 16.2. A single WRSF design is planned for Florida Mountain along with two backfill dumps into the Florida Mountain phase 1 and 2 pits. Material from Florida Mountain phase 1 will be placed into the primary WRSF. Phase 2 waste material will also be placed into the primary WRSF except for some upper areas of the pit where some waste will be backfilled. Phase 3 waste material is planned to be placed into the backfill dump as available while the remaining waste material will be placed into the Florida Mountain WRSF. The total capacity of the WRSF is 32.2 million cubic meters (42.1 million cubic yards). The remaining 23.4 million cubic meters (30.6 million cubic yards) of waste material will be placed into backfill.

Three WRSF designs were created for the DeLamar area which includes a West WRSF, East WRSF, and a North WRSF. The West and East WRSFs are intended for storage of material from the DeLamar Main phase 1 pit. Both dump designs include a roadway that will be built into the WRSFs to allow haulage through the main pit exits for both DeLamar Main and Sullivan Gulch pits. The East WRSF creates its haulage road through a valley to the south of the deeper Sullivan Gulch phase 2 pit. This road is anticipated to be in place well before the mining of Sullivan Gulch phase 2. The total West DeLamar WRSF total capacity is 5.9 million cubic meters (7.7 million cubic yards). After the roadway is completed, the East WRSF is to be expanded to the south. The total East DeLamar WRSF total capacity will be 50.0 million cubic meters (65.4 million cubic yards).

The North WRSF will be located in a valley to the north of the Main and Sullivan Gulch pits. This will be used for the Main pit phase 2 waste along with Sullivan Gulch pit waste. The designed capacity of the North WRSF is 26.4 million cubic meters (34.5 million cubic yards). As available, additional waste will be placed into the Main phase 1 pit and from the Main phase 2 pit as backfill. Additional backfill material will be placed into the Main phase 2 pit from Sullivan Gulch phase 1 mining.

Table 16-2 WRSF and Backfill Design Capacities

	K Cu M	K Tonnes
Florida Mountain WRSF	32,160	54,425
Florida Mountain Phase 1 Bckfl	4,416	7,473
Florida Mountain Ultimate Bckfl	19,447	32,911
Florida Mountain WRSF Total	**56,023**	**94,809**
DeLamar West WRSF	5,859	9,915
DeLamar East WRSF	50,000	84,615
DeLamar North WRSF	26,410	44,694
Main P1 Bckfl	1,498	2,535
Main Ult Bckfl	17,361	29,380
Sullivan Bckfl	8,283	14,018
DeLamar WRSF Total	**109,411**	**185,158**
Project WRSF Total	**165,435**	**279,966**

16.2 Mine Production Schedule

Production scheduling was completed using Geovia's MineSched™ (version 2021) software. Proven and Probable reserves along with waste material inside pit designs previously discussed were used to schedule mine production.

The production schedule considers the processing of Florida Mountain oxide and mixed material by crushing and heap leaching. Florida Mountain non-oxide material would be processed using flotation followed by cyanide leaching of the flotation concentrate. Processing of the DeLamar material will require crushing and agglomeration prior to heap leaching and non-oxide material will be processed through the mill.

Monthly periods were used to create the production schedule with pre-stripping starting in Florida Mountain at month -5. The start of leach processing is scheduled in month 1, though a total of 610,000 tonnes of leach material is to be mined during preproduction. It is assumed that this material will be crushed and used as overliner on top of the leach pad liner. The nominal rate for leach processing will be 35,000 tonnes per day or 12,600,000 tonnes per year. Note that during the first year, a total of 10,287,000 tonnes will be processed along with the material laid onto the pad during preproduction. This represents a ramp up to full processing.

The leach pad stream of material will not always maximize the process. Full throughput is dependent on availability of leach ore.

Leaching starts with Florida Mountain material in month 1 and DeLamar leach material is processed starting in year 2. Prior to that, the agglomeration circuit will be installed. The DeLamar leach material will be processed up to the same rate as the Florida Mountain material.

Florida Mountain and DeLamar non-oxide material will be stockpiled until the flotation mill is constructed. The start of the 6,000 tonne per day mill will be in year 3 with 1,982,000 tonnes processed in that year, increasing to 2,160,000 tonnes per year after that until the non-oxide 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 six months. A maximum of 138,000 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 Florida Mountain and DeLamar is summarized in Table 16.3 and Table 16.4, respectively. Table 16.5 summarizes the total yearly mine production schedule.

Table 16-3 Florida Mountain Mine Production Schedule

		Units	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Total
Florida Mountain Total	Leach Mined	K Tonnes	944	10,041	12,356	7,711	11,162	3,471	22	-	45,708
		g Au/t	0.35	0.51	0.42	0.41	0.41	0.38	0.27	-	0.43
		K Ozs Au	11	163	169	102	148	43	0	-	635
		g Ag/t	8.91	13.61	9.88	8.55	11.71	16.32	22.80	-	11.40
		K Ozs Ag	271	4,395	3,924	2,120	4,203	1,821	16	-	16,749
	Pit to Mill StkPl	K Tonnes	2	1,167	406	1,787	1,696	1,556	8	-	6,622
		g Au/t	0.43	0.77	0.40	0.36	0.29	0.28	0.27	-	0.40
		K Ozs Au	0	29	5	21	16	14	0	-	85
		g Ag/t	2.01	7.76	12.15	6.97	8.17	8.01	10.59	-	7.98
		K Ozs Ag	0	291	159	400	445	401	3	-	1,699
	Pit to Mill	K Tonnes	-	-	-	706	1,659	1,559	9	-	3,933
		g Au/t	-	-	-	0.81	0.78	0.70	0.50	-	0.75
		K Ozs Au	-	-	-	18	41	35	0	-	95
		g Ag/t	-	-	-	13.27	20.21	21.49	22.53	-	19.48
		K Ozs Ag	-	-	-	301	1,078	1,077	6	-	2,463
	Total Mined Above COG	K Tonnes	946	11,209	12,762	10,204	14,516	6,587	39	-	56,263
		g Au/t	0.35	0.53	0.42	0.43	0.44	0.43	0.32	-	0.45
		K Ozs Au	11	192	174	140	205	92	0	-	815
		g Ag/t	8.89	13.00	9.95	8.60	12.27	15.58	20.31	-	11.56
		K Ozs Ag	271	4,686	4,082	2,822	5,726	3,300	25	-	20,911
	Ox_Wst	K Tonnes	2,438	8,196	5,706	6,398	4,869	2,043	-	-	29,650
	Mx_Wst	K Tonnes	1,038	4,429	8,243	5,831	11,366	4,239	18	-	35,165
	Su_Wst	K Tonnes	12	860	1,434	8,420	9,337	6,650	22	-	26,735
	Min_Wst	K Tonnes	-	-	40	60	33	-	-	-	132
	Fill_Wst	K Tonnes	2	14	37	-	2,308	-	-	-	2,360
	Total Waste	K Tonnes	3,491	13,498	15,459	20,710	27,912	12,933	40	-	94,043
	Total Mined	K Tonnes	4,437	24,707	28,221	30,914	42,429	19,520	79	-	150,306
	Strip Ratio	W:O	3.69	1.20	1.21	2.03	1.92	1.96	1.03		1.67

COG = cutoff grade.

The material sent to the crusher for heap-leach processing was scheduled by RESPEC based on the mine production schedule and metal recoveries. The estimated heap-leach recovered ounces were modeled by Integra's metallurgical process consultant, Mr. Michael Botz, as reported in Section 17.0. The recoveries used to estimate recoverable gold are those shown in Table 15.2. Leach K Ozs Au Rec and K Ozs Ag Rec in Table 16.6 shows recoverable ounces based on the recoveries provided. The yearly process production summary in Table 16.6 also shows the tonnage, grade, contained ounces of silver and gold, and the recovered silver and gold ounces from mill processing.

Table 16-4 DeLamar PFS 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
DeLamar Pits Total	Leach Mined	K Tonnes	-	-	2,220	2,893	79	9,525	13,940	8,901	6,325	997	296	450	259	176	170	200	103	0	-	46,533
		g Au/t	-	-	0.41	0.47	0.30	0.35	0.36	0.34	0.29	0.19	0.46	0.60	0.70	0.79	0.74	0.61	0.48	0.78	-	0.36
		K Ozs Au	-	-	30	44	1	107	163	98	59	6	4	9	6	4	4	4	2	0	-	540
		g Ag/t	-	-	20.22	31.41	12.10	12.50	17.91	23.90	33.16	44.62	24.02	57.84	77.92	77.57	78.07	74.82	71.64	143.50	-	23.10
		K Ozs Ag	-	-	1,444	2,922	31	3,827	8,026	6,841	6,744	1,431	229	837	648	439	426	480	236	0	-	34,560
	Pit to Mill StkPl	K Tonnes	-	-	30	752	41	57	564	574	905	561	443	536	342	297	320	296	352	4	-	6,074
		g Au/t	-	-	0.48	0.43	0.42	0.55	0.69	0.40	0.39	0.31	0.40	0.41	0.41	0.40	0.41	0.43	0.42	0.37	-	0.43
		K Ozs Au	-	-	0	10	1	1	13	7	11	6	6	7	5	4	4	4	5	0	-	83
		g Ag/t	-	-	38.25	38.78	22.48	23.30	15.90	22.08	25.49	32.35	20.70	21.44	20.88	21.51	21.21	20.21	20.90	22.16	-	24.67
		K Ozs Ag	-	-	37	938	29	43	288	408	741	583	295	370	229	206	218	192	237	3	-	4,817
	Pit to Mill	K Tonnes	-	-	-	883	14	79	691	965	1,258	866	681	1,454	1,402	1,559	1,546	1,612	1,556	49	-	14,614
		g Au/t	-	-	-	0.53	0.25	1.18	1.13	0.57	0.48	0.67	0.52	0.65	0.82	0.86	0.95	0.84	0.73	0.53	-	0.74
		K Ozs Au	-	-	-	15	0	3	25	18	19	19	11	30	37	43	47	43	36	1	-	349
		g Ag/t	-	-	-	77.77	44.88	91.34	37.85	65.18	66.43	62.05	63.88	73.92	72.86	76.77	75.59	67.02	61.66	47.93	-	68.35
		K Ozs Ag	-	-	-	2,208	20	231	841	2,023	2,686	1,727	1,398	3,456	3,285	3,849	3,758	3,474	3,084	76	-	32,114
	Total Mined Above COG	K Tonnes	-	-	2,250	4,529	133	9,661	15,194	10,440	8,487	2,423	1,420	2,441	2,003	2,033	2,037	2,108	2,011	53	-	67,221
		g Au/t	-	-	0.41	0.47	0.33	0.36	0.41	0.37	0.33	0.39	0.47	0.59	0.74	0.78	0.85	0.76	0.66	0.52	-	0.45
		K Ozs Au	-	-	30	69	1	111	201	123	89	30	21	46	47	51	56	51	43	1	-	972
		g Ag/t	-	-	20.46	41.68	18.63	13.20	18.74	27.62	37.27	48.01	42.09	59.42	64.64	68.76	67.24	61.19	55.04	46.00	-	33.08
		K Ozs Ag	-	-	1,480	6,068	80	4,101	9,155	9,271	10,171	3,741	1,921	4,663	4,162	4,494	4,403	4,146	3,558	79	-	71,491
	Ox_Wst	K Tonnes	-	-	3,892	481	33	4,630	4,723	10,361	2,536	16,042	8,488	1,900	126	-	-	-	-	-	-	53,234
	Mx_Wst	K Tonnes	-	-	156	594	28	2,275	2,373	5,049	9,223	4,917	6,579	3,626	1,384	541	33	-	0	-	-	36,777
	Su_Wst	K Tonnes	-	-	43	856	19	1,135	3,398	2,384	9,941	9,077	15,740	9,267	6,857	4,659	2,919	1,459	479	7	-	68,239
	Min_Wst	K Tonnes	-	-	15	356	11	97	789	574	1,054	390	623	753	416	303	293	258	252	4	-	6,189
	Fill_Wst	K Tonnes	-	-	-	-	22	1,191	7,119	4,814	1,589	0	-	-	-	-	-	-	-	-	-	14,735
	Total Waste	K Tonnes	-	-	4,107	2,287	113	9,328	18,403	23,182	24,363	30,427	31,430	15,546	8,782	5,503	3,246	1,717	731	11	-	179,174
	Total Mined	K Tonnes	-	-	6,357	6,815	246	18,989	33,597	33,621	32,850	32,850	32,850	17,987	10,785	7,535	5,282	3,825	2,742	64	-	246,395
	Strip Ratio	W:O			1.83	0.50	0.85	0.97	1.21	2.22	2.87	12.56	22.14	6.37	4.39	2.71	1.59	0.81	0.36	0.21		2.67

Table 16-5 Total PFS 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Total Project	Leach Mined	K Tonnes	944	10,041	14,576	10,604	11,240	12,996	13,962	8,901	6,325	997	296	450	259	176	170	200	103	0	-	92,241
		g Au/t	0.35	0.51	0.42	0.43	0.41	0.36	0.36	0.34	0.29	0.19	0.46	0.60	0.70	0.79	0.74	0.61	0.48	0.78	-	0.40
		K Ozs Au	11	163	198	145	149	150	164	98	59	6	4	9	6	4	4	4	2	0	-	1,175
		g Ag/t	8.91	13.61	11.45	14.79	11.71	13.52	17.92	23.90	33.16	44.62	24.02	57.84	77.92	77.57	78.07	74.82	71.64	143.50	-	17.30
		K Ozs Ag	271	4,395	5,367	5,042	4,233	5,648	8,042	6,841	6,744	1,431	229	837	648	439	426	480	236	0	-	51,310
	Pit to Mill StkPl	K Tonnes	2	1,167	436	2,539	1,737	1,613	571	574	905	561	443	536	342	297	320	296	352	4	-	12,695
		g Au/t	0.43	0.77	0.41	0.38	0.30	0.29	0.69	0.40	0.39	0.31	0.40	0.41	0.41	0.40	0.41	0.43	0.42	0.37	-	0.41
		K Ozs Au	0	29	6	31	17	15	13	7	11	6	6	7	5	4	4	4	5	0	-	168
		g Ag/t	2.01	7.76	13.94	16.39	8.50	8.55	15.83	22.08	25.49	32.35	20.70	21.44	20.88	21.51	21.21	20.21	20.90	22.16	-	15.96
		K Ozs Ag	0	291	195	1,338	475	444	291	408	741	583	295	370	229	206	218	192	237	3	-	6,516
	Pit to Mill	K Tonnes	-	-	-	1,589	1,672	1,638	700	965	1,258	866	681	1,454	1,402	1,559	1,546	1,612	1,556	48	-	18,547
		g Au/t	-	-	-	0.65	0.77	0.73	1.12	0.57	0.48	0.67	0.52	0.65	0.82	0.86	0.95	0.84	0.73	0.53	-	0.74
		K Ozs Au	-	-	-	33	42	38	25	18	19	19	11	30	37	43	47	43	36	1	-	444
		g Ag/t	-	-	-	49.11	20.41	24.85	37.66	65.18	66.43	62.05	63.88	73.92	72.86	76.77	75.59	67.02	61.66	47.93	-	57.99
		K Ozs Ag	-	-	-	2,509	1,097	1,309	847	2,023	2,686	1,727	1,398	3,456	3,285	3,849	3,758	3,474	3,084	76	-	34,577
	Total Mined Above COG	K Tonnes	946	11,209	15,012	14,733	14,649	16,247	15,233	10,440	8,487	2,423	1,420	2,441	2,003	2,033	2,037	2,108	2,011	53	-	123,483
		g Au/t	0.35	0.53	0.42	0.44	0.44	0.39	0.41	0.37	0.33	0.39	0.47	0.59	0.74	0.78	0.85	0.76	0.66	0.52	-	0.45
		K Ozs Au	11	192	204	209	207	203	201	123	89	30	21	46	47	51	56	51	43	1	-	1,787
		g Ag/t	8.89	13.00	11.52	18.77	12.33	14.17	18.74	27.62	37.27	48.01	42.09	59.42	64.64	68.76	67.24	61.19	55.04	46.00	-	23.27
		K Ozs Ag	271	4,686	5,563	8,890	5,805	7,400	9,180	9,271	10,171	3,741	1,921	4,663	4,162	4,494	4,403	4,146	3,558	79	-	92,403
	Ox_Wst	K Tonnes	2,488	8,196	9,598	6,879	4,903	6,673	4,723	10,361	2,556	16,042	8,488	1,900	126	-	-	-	-	-	-	82,884
	Mx_Wst	K Tonnes	1,088	4,429	8,399	6,425	11,394	6,515	2,391	5,049	9,223	4,917	6,579	3,626	1,384	541	33	-	0	-	-	71,941
	Min_Wst	K Tonnes	12	860	1,477	9,277	9,356	7,785	3,419	2,384	9,941	9,077	15,740	9,267	6,857	4,659	2,919	1,459	479	7	-	94,974
	Un_Wst	K Tonnes	-	-	55	416	43	97	789	574	1,054	390	623	753	416	303	293	258	252	4	-	6,322
	Fill_Wst	K Tonnes	2	14	37	-	2,329	1,191	7,119	4,814	1,589	0	-	-	-	-	-	-	-	-	-	17,096
	Total Waste	K Tonnes	3,491	13,498	19,566	22,996	28,025	22,261	18,443	23,182	24,363	30,427	31,430	15,546	8,782	5,503	3,246	1,717	731	11	-	273,217
	Total Mined	K Tonnes	4,437	24,707	34,578	37,729	42,674	38,508	33,676	33,621	32,850	32,850	32,850	17,987	10,785	7,535	5,282	3,825	2,742	64	-	396,701
	Strip Ratio	W:O	3.69	1.20	1.30	1.56	1.91	1.37	1.21	2.22	2.87	12.56	22.14	6.37	4.39	2.71	1.59	0.81	0.36	0.21		2.21

Table 16-6 PFS 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	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Total
Florida Mnt Leach	K Tonnes	610	10,287	10,596	9,474	10,993	3,386	15	348	-	-	-	-	-	-	-	-	-	-	-	45,708
	g Au/t	0.42	0.50	0.46	0.37	0.42	0.39	0.33	0.16	-	-	-	-	-	-	-	-	-	-	-	0.43
	K Ozs Au	8	165	158	112	147	43	0	2	-	-	-	-	-	-	-	-	-	-	-	635
	K Ozs Au Rec	-	132	124	85	111	32	0	1	-	-	-	-	-	-	-	-	-	-	-	485
	g Ag/t	10.55	13.43	10.44	8.16	11.76	16.20	24.54	11.22	-	-	-	-	-	-	-	-	-	-	-	11.40
	K Ozs Ag	207	4,443	3,556	2,484	4,158	1,764	12	125	-	-	-	-	-	-	-	-	-	-	-	16,749
	K Ozs Ag Rec	-	2,205	1,688	1,184	1,968	834	6	59	-	-	-	-	-	-	-	-	-	-	-	7,944
DeLamar Leach	K Tonnes	-	-	1,953	3,161	79	9,214	12,585	10,566	6,325	997	296	450	259	176	170	200	103	0	-	46,533
	g Au/t	-	-	0.44	0.45	0.30	0.35	0.38	0.32	0.29	0.19	0.46	0.60	0.70	0.79	0.74	0.61	0.48	0.78	-	0.36
	K Ozs Au	-	-	28	45	1	105	155	108	59	6	4	9	6	4	4	4	2	0	-	540
	K Ozs Au Rec	-	-	20	26	0	71	105	74	38	3	3	6	4	3	3	2	1	0	-	359
	g Ag/t	-	-	20.85	30.08	12.10	12.71	18.50	21.91	33.16	44.62	24.02	57.84	77.92	77.57	78.07	74.82	71.64	143.50	-	23.10
	K Ozs Ag	-	-	1,309	3,057	31	3,764	7,486	7,443	6,744	1,431	229	837	648	439	426	480	236	0	-	34,560
	K Ozs Ag Prod	-	-	235	549	13	930	2,426	2,651	2,543	593	87	321	252	171	166	187	92	0	-	11,216
Total Leach	K Tonnes	610	10,287	12,548	12,635	11,072	12,600	12,600	10,914	6,325	997	296	450	259	176	170	200	103	0	-	92,241
	g Au/t	0.42	0.50	0.46	0.39	0.42	0.36	0.38	0.31	0.29	0.19	0.46	0.60	0.70	0.79	0.74	0.61	0.48	0.78	-	0.40
	K Ozs Au	8	165	186	158	148	147	155	110	59	6	4	9	6	4	4	4	2	0	-	1,175
	K Ozs Au Rec	-	132	144	111	112	103	105	75	38	3	3	6	4	3	3	2	1	0	-	844
	g Ag/t	10.55	13.43	12.06	13.64	11.77	13.65	18.51	21.57	33.16	44.62	24.02	57.84	77.92	77.57	78.07	74.82	71.64	143.50	-	17.30
	K Ozs Ag	207	4,443	4,865	5,541	4,188	5,528	7,498	7,568	6,744	1,431	229	837	648	439	426	480	236	0	-	51,310
	K Ozs Ag Rec	-	2,205	1,923	1,733	1,981	1,764	2,431	2,710	2,543	593	87	321	252	171	166	187	92	0	-	19,160
Florida Mountain Mill	K Tonnes	-	-	-	1,078	1,885	1,863	926	558	445	492	740	346	379	300	307	277	302	656	-	10,555
	g Au/t	-	-	-	1.09	0.77	0.67	0.38	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	-	0.53
	K Ozs Au	-	-	-	38	46	40	11	5	4	5	7	3	3	3	3	3	3	6	-	180
	K Ozs Au Rec	-	-	-	33	40	34	9	4	3	3	5	2	3	2	2	2	2	5	-	149
	g Ag/t	-	-	-	15.57	19.35	19.56	8.00	6.74	6.74	6.74	6.74	6.74	6.74	6.74	6.74	6.74	6.74	6.74	-	12.26
	K Ozs Ag	-	-	-	540	1,173	1,171	238	121	96	107	160	75	82	65	66	60	65	142	-	4,162
	K Ozs Ag Rec	-	-	-	396	873	874	164	81	64	71	107	50	55	43	44	40	44	95	-	3,002
DeLamar Mill	K Tonnes	-	-	-	904	275	297	1,234	1,608	1,715	1,668	1,420	1,819	1,781	1,860	1,853	1,889	1,858	506	-	20,688
	g Au/t	-	-	-	0.54	0.47	0.62	0.89	0.54	0.46	0.52	0.43	0.60	0.74	0.78	0.86	0.78	0.68	0.43	-	0.65
	K Ozs Au	-	-	-	16	4	6	35	28	25	28	20	35	42	47	51	47	40	7	-	432
	K Ozs Au Rec	-	-	-	6	2	2	11	10	10	11	7	13	16	18	20	18	15	3	-	161
	g Ag/t	-	-	-	77.47	58.17	53.40	28.74	47.40	54.19	47.50	42.39	63.44	61.76	67.77	66.58	60.26	55.00	23.45	-	55.52
	K Ozs Ag	-	-	-	2,252	515	510	1,140	2,451	2,988	2,547	1,936	3,711	3,537	4,052	3,966	3,659	3,285	382	-	36,931
	K Ozs Ag Rec	-	-	-	1,939	448	391	793	2,102	2,226	1,998	1,424	2,709	2,582	2,958	2,896	2,671	2,398	279	-	27,834
Total Mill	K Tonnes	-	-	-	1,982	2,160	2,160	2,160	2,166	2,160	2,160	2,160	2,166	2,160	2,160	2,160	2,166	2,160	1,162	-	31,243
	g Au/t	-	-	-	0.84	0.73	0.67	0.67	0.48	0.43	0.47	0.38	0.55	0.66	0.72	0.78	0.71	0.62	0.35	-	0.61
	K Ozs Au	-	-	-	53	51	46	47	33	30	33	27	38	46	50	54	50	43	13	-	612
	K Ozs Au Rec	-	-	-	39	41	36	20	14	13	14	13	16	19	20	22	20	18	7	-	311
	g Ag/t	-	-	-	43.80	24.30	24.21	19.85	36.94	44.41	38.21	30.18	54.37	52.11	59.28	58.07	53.41	48.25	14.01	-	40.91
	K Ozs Ag	-	-	-	2,792	1,687	1,681	1,379	2,572	3,084	2,653	2,096	3,786	3,619	4,117	4,033	3,719	3,351	524	-	41,093
	K Ozs Ag Rec	-	-	-	2,356	1,320	1,265	957	2,183	2,291	2,069	1,531	2,759	2,637	3,001	2,940	2,711	2,442	374	-	30,836
Total Project	K Tonnes	610	10,287	12,548	14,617	13,232	14,760	14,760	13,080	8,485	3,157	2,456	2,616	2,419	2,336	2,330	2,366	2,263	1,162	-	123,483
	g Au/t	0.42	0.50	0.46	0.45	0.47	0.41	0.43	0.34	0.32	0.38	0.39	0.56	0.66	0.72	0.78	0.70	0.62	0.35	-	0.45
	K Ozs Au	8	165	186	211	198	194	202	148	88	39	31	47	51	54	58	54	45	13	-	1,787
	K Ozs Au Rec	-	132	144	150	153	139	125	89	50	17	16	22	22	23	24	22	19	7	-	1,154
	g Ag/t	10.55	13.43	12.06	17.73	13.81	15.19	18.71	24.11	36.03	40.23	29.44	54.96	54.87	60.66	59.53	55.22	49.31	14.01	-	23.27
	K Ozs Ag	207	4,443	4,865	8,333	5,876	7,209	8,877	10,141	9,828	4,084	2,325	4,623	4,267	4,556	4,459	4,200	3,587	524	-	92,403
	K Ozs Ag Rec	-	2,205	1,923	4,088	3,302	3,029	3,389	4,898	4,833	2,662	1,618	3,080	2,889	3,172	3,106	2,898	2,534	374	-	49,996
	K AuEq Ozs	-	160	168	202	195	178	167	151	111	51	36	60	59	63	64	59	51	12	-	1,787

Stockpiles will be located near the Railveyor loadout facilities. The stockpiles will be used to store low-grade material longer term, as well as some higher-grade material during initial mining. Stockpile management will be required not only to be able to manage the metal grades to be processed but will also be critical to manage blending of the various ore types. Stockpile management will require additional studies in the future to ensure optimization of the mine.

Table 16.7 and Table 16.8 show stockpile balance sheets for the heap-leach and mill material, respectively.

Table 16-7 Leach Ore Stockpile Balance

Leach Stockpiles	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Yr_12	Yr_13
Added to StkPl	K Tonnes	417	1,009	2,173	1,194	621	1,032	1,380	0	0	0	0	-	0	-
	g Au/t	0.30	0.24	0.19	0.20	0.23	0.19	0.20	0.18	0.16	0.08	0.21	-	0.20	-
	K Ozs Au	4	8	14	8	5	6	9	0	0	0	0	-	0	-
	g Ag/t	6.00	7.33	7.62	9.36	8.10	8.79	10.77	14.65	15.06	24.95	4.66	-	11.81	-
	K Ozs Ag	80	238	532	359	162	292	478	0	0	0	0	-	0	-
Removed from StkPl	K Tonnes	82	1,254	145	3,224	452	635	19	2,014	-	0	-	-	-	-
	g Au/t	0.65	0.24	0.24	0.20	0.25	0.19	0.28	0.20	-	-	-	-	-	-
	K Ozs Au	2	10	1	20	4	4	0	13	-	-	-	-	-	-
	g Ag/t	6.33	7.01	7.82	8.26	8.07	8.84	10.40	10.13	-	-	-	-	-	-
	K Ozs Ag	17	283	36	856	117	181	6	656	-	-	-	-	-	-
StkPl Balance	K Tonnes	334	89	2,117	86	255	652	2,014	-	-	-	-	-	-	-
	g Au/t	0.21	0.18	0.19	0.18	0.18	0.19	0.20	-	-	-	-	-	-	-
	K Ozs Au	2	1	13	1	1	4	13	-	-	-	-	-	-	-
	g Ag/t	5.92	6.46	7.56	6.21	7.51	8.24	9.96	-	-	-	-	-	-	-
	K Ozs Ag	64	18	514	17	62	173	645	-	-	-	-	-	-	-

Table 16-8 Mill Ore Stockpile Balance

Mill Stockpiles	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17
Added to Stk Pl	K Tonnes	2	1,167	436	2,539	1,737	1,613	228	370	742	555	206	242	10	21	23	36	56	-
	gAu/t	0.43	0.77	0.41	0.38	0.30	0.29	0.72	0.43	0.39	0.31	0.41	0.41	0.47	0.41	0.42	0.43	0.43	-
	K Ozs Au	0	29	6	31	17	15	5	5	9	6	3	3	0	0	0	0	1	-
	g Ag/t	2.01	7.76	13.94	16.39	8.50	8.55	16.98	23.05	26.36	32.42	20.31	22.42	18.40	21.10	20.91	20.40	20.88	-
	K Ozs Ag	0	291	195	1,338	475	444	124	274	628	578	135	175	6	14	16	24	37	-
Removed from Stk Pl	K Tonnes	-	-	-	393	488	522	1,116	997	739	1,288	1,243	417	426	325	317	294	308	1,109
	gAu/t	-	-	-	1.59	0.57	0.48	0.41	0.38	0.35	0.34	0.31	0.31	0.30	0.29	0.29	0.29	0.29	0.34
	K Ozs Au	-	-	-	20	9	8	15	12	8	14	12	4	4	3	3	3	3	12
	g Ag/t	-	-	-	22.36	37.62	22.22	9.73	13.52	12.04	22.23	13.32	10.13	8.28	7.77	7.16	7.56	7.00	12.48
	K Ozs Ag	-	-	-	283	590	373	349	433	286	921	532	136	113	81	73	72	69	445
Stk Pl Balance	K Tonnes	2	1,169	1,605	3,751	5,000	6,091	5,202	4,575	4,578	3,844	2,807	2,632	2,216	1,913	1,619	1,361	1,109	-
	gAu/t	0.43	0.77	0.67	0.38	0.33	0.31	0.30	0.30	0.30	0.29	0.29	0.30	0.30	0.31	0.31	0.32	0.34	-
	K Ozs Au	0	29	35	46	53	60	51	44	45	36	27	26	22	19	16	14	12	-
	g Ag/t	2.01	7.75	9.43	12.79	8.88	7.65	7.61	7.57	9.89	9.01	7.93	8.91	9.08	9.43	10.04	10.86	12.43	-
	K Ozs Ag	0	291	487	1,542	1,427	1,498	1,273	1,114	1,456	1,113	716	754	647	580	523	475	443	-

16.3 Equipment Requirements

The PFS assumed owner mining instead of the more expensive contract mining. The production schedule was used along with additional efficiency factors, performance curves, and productivity rates to develop the first-principal 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 16.9 shows the yearly equipment requirements.

Table 16-9 PFS 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Max
Production Drills	#	3	4	5	5	5	5	5	5	4	4	4	4	3	2	2	1	1	1	-	5
Pioneering Drills	#	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1
Loader	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Hydraulic Shovel	#	1	2	2	2	2	2	2	2	2	2	2	2	1	-	-	-	-	-	-	2
Haul Trucks	#	6	12	15	16	16	16	16	11	11	11	11	11	7	5	3	3	3	2	-	16
Support Equipment																					
D10 Type Dozer	#	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	-	2
D9 Type Dozer	#	1	1	1	1	1	1	1	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	1	1	1	1	1	1	-	2
Water Truck - 20,000 gal	#	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	-	2
Pit Pumps	#	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	-	2
50 Ton Crane	#	1	1	1	1	1	1	1	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	1	1	1	1	1	1	-	1
Blasting																					
Skid Loader	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Stemming Truck	#	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Explosives Truck	#	1	1	1	1	1	1	1	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	1	1	1	1	1	1	-	1
Mechanic/Service Truck	#	1	2	2	2	2	2	2	2	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	1	1	1	1	1	1	-	1
Other Mine Equipment																					
Light Plants	#	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	4

Note that earlier mining studies identified the water table to be around the 1,810-meter (5,938 foot) elevation. All of the Florida Mountain mining and most the DeLamar mining is above this elevation. The mining at Sullivan Gulch phase 2 does extend about 160 meters (525 feet) below the 1,810 elevation. It is assumed that the two pit pumps will be sufficient to maintain a dry pit. Additional studies will be completed as part of a feasibility study.

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 3.0 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 16.10.

Table 16-10 Schedule Efficiency

Daily Schedule	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 26.3, 27.5, 30.3 meters per hour (86.3, 90.2, 99.4 feet 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, one pioneer drill and five production drills are estimated to be needed. It is assumed that these drills will last through the life of mine ("LOM") with an availability of 85% for the life of the drill.

Loading equipment is anticipated to include one large 13-cubic meter (17 cubic yard) loader and two 23-cubic meter (30 cubic yard) hydraulic shovels. The loader 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 loading units. No replacement loaders were assumed. The overall use of available hours is 33%.

Two hydraulic shovels are used as the primary loading tool. The initial shovel starts operating in month -6 and the second shovel starts working in month 1. 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 80%.

Haul trucks are assumed to be 136-tonne capacity rigid frame trucks. Haulage hours were developed using MineSched software (Version 2021). MineSched uses 3-dimensional centerlines drawn for bench, in-pit, and ex-pit travel. The performance and retard curve data are input into the software, and MineSched uses that along with the truck capacity and load, dump, and spot times to determine the time required to haul material to its destination. The hours developed from MineSched are considered productive hours, and these are adjusted in the mining cost spreadsheets to include an 83% efficiency.

The loading time provided in the software is based on the hydraulic shovel and is included in the productive hour calculation. This is adjusted in spreadsheets to reflect the use of loaders; thus, the load time is dependent 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 2.70 minutes. Spot time at the loader or shovel was 0.50 minutes and the spot and dump time was a combined 1.20 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 16 haul trucks are purchased to maintain the production schedule. This assumes a 1% per year declining availability from 90% down to 85%.

Railveyor will primarily be used for haulage of ore material from stockpiles near the pits to the crusher feeding both the heap-leach pad and the mill. Discussion on Railveyor along with the layout is provided in Section 18.7.

16.4 Personnel Requirements

Table 16.11 shows the estimated mine operations personnel requirements (full mine site personnel requirements are shown in Table 16.11). 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. The peak mining personnel requirement is 250 people on an annual basis.

Table 16-11 PFS Mining Personnel Requirements

Administration	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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Max
Project/Construction Manager	#	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1
Project Manager Assistant	#	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1
Mine General Manager	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Administrative Assistant	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Administrative Superintendent / Controller	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Human Resources	#	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	2
Accounts Receivable / Payable	#	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	-	2
Purchasing Agent	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Warehousemen	#	3	4	4	4	4	4	4	4	4	4	4	4	2	1	1	1	1	1	-	4
Janitor	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Safety and Security Superintendent	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Safety Specialist	#	3	4	4	4	4	4	4	4	4	4	4	4	2	1	1	1	1	1	-	4
Security Guard	#	3	4	4	4	4	4	4	4	4	4	4	4	2	1	1	1	1	1	-	4
Environmental Superintendent	#	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Environmental Specialist	#	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	2
Total Administration	#	24	27	24	24	24	24	24	24	24	24	24	24	17	14	14	14	14	14	-	27
Mining General Personnel																					
Mine Superintendent	#	1	1	1	1	1	1	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	1	1	1	1	1	1	-	1
Mine Foremen	#	4	4	4	4	4	4	4	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	1	1	1	1	1	-	-	1
Mine Engineer	#	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	-	2
Chief Surveyor	#	1	1	1	1	1	1	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	1	1	1	1	1	1	-	3
Chief Geologist	#	1	1	1	1	1	1	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	2	2	2	2	2	2	-	4
Samplers	#	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	-	4
Total Mine General	#	22	22	22	22	22	22	22	22	22	22	22	22	15	15	15	15	15	11	-	22
Mine Operations Hourly Personnel																					
Operators																					
Blasters	#	2	2	2	2	2	2	2	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	2	2	2	2	2	2	-	2
Drill Operators	#	12	16	20	20	20	20	20	20	16	16	16	16	12	8	8	4	4	4	-	20
Loader Operators	#	8	12	12	12	12	12	12	12	10	10	10	10	6	2	2	2	2	2	-	12
Haul Truck Operators	#	24	48	60	64	64	64	64	44	44	44	44	44	28	20	12	12	12	8	-	64
Support Equipment Operators	#	12	17	17	17	17	17	17	17	17	17	17	17	10	10	10	10	10	10	-	17
General Mine Labors	#	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Operators	#	60	97	113	117	117	117	117	97	91	91	91	91	60	44	36	32	32	28	-	117
Mechanics																					
Mechanics - Drilling	#	6	8	10	10	10	10	10	10	8	8	8	8	6	4	4	2	2	2	-	10
Mechanics - Loading	#	6	8	10	10	10	10	10	10	8	8	8	8	6	4	4	2	2	2	-	10
Mechanics - Haulage	#	12	24	30	30	30	30	30	22	22	22	22	22	14	10	6	6	6	4	-	30
Mechanics - Support	#	6	9	9	9	9	9	9	9	9	9	9	9	5	5	5	5	5	5	-	9
Total Mechanics	#	30	49	59	59	59	59	59	51	47	47	47	47	31	23	19	15	15	13	-	59
Maintenance																					
Maintenance Superintendent	#	1	1	1	1	1	1	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	4	4	4	4	4	4	-	4
Maintenance Planners	#	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	-	2
Light Vehicle Mechanic	#	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	-	2
Welder	#	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	-	4
Servicemen	#	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	-	4
Tireman	#	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	-	4
Maintenance Labor	#	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	-	4
Total Maintenance	#	25	25	25	25	25	25	25	25	25	25	25	25	15	15	15	15	15	14	-	25

17.0 RECOVERY METHODS

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

- Lower-grade oxide and mixed materials will be processed by crushed-ore cyanide heap leaching; and
- Non-oxide material will be processed using grinding followed by flotation, and very fine grinding of flotation concentrate for agitated cyanide leaching.

Heap-leach and milling ores will be coming from both the Florida Mountain and DeLamar deposits. Pregnant solutions from the heap-leach operation and from the milling operation will be processed by the same Merrill-Crowe zinc cementation plant. Processing will start with heap leaching in the first two years of operation. Milling of higher-grade non-oxide ore will start in the third year of operation.

Both Florida Mountain and DeLamar oxide and mixed ore types have been shown to be amenable to heap-leach processing following crushing. Material will be crushed in three stages to a nominal size of 80% finer than (P_{80}) 12.7-millimeter (0.5 inches), at a rate of 35,000 tonnes per day. About 45% of DeLamar ore is expected to require agglomeration.

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. Pregnant 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 precipitate will be filtered, dried, and smelted to produce gold and silver doré bullion for shipment off site.

The milling process will start with primary crushing of the ore to a nominal P_{80} of 120 millimeter (4.72 inches), followed by grinding in a SAG mill-ball mill circuit to a P_{80} of 150 microns. The ball mill discharge will be pumped to hydrocyclones, with the hydrocyclone overflow advancing to flotation and the underflow returning to the ball mill.

The flotation circuit will produce a sulfide concentrate that will recover gold and silver from the ore. This flotation concentrate will be reground to a nominal P_{80} of 20 microns before being leached in agitated leach tanks. Pregnant solution will be separated using a CCD circuit that employs dewatering cyclones and thickeners. The pregnant solution is then sent to the Merrill-Crowe plant and gold smelting facility to produce gold and silver doré bullion.

The flotation tailing stream will be thickened and pumped to the tailing storage facility. The concentrate leach residue will be sent to cyanide destruction, then stored in a separate concentrate leach tailing storage facility.

17.1 Process Production Schedule

The process facilities for the DeLamar project will be developed to accommodate the mining sequences of the Florida Mountain and DeLamar deposits as summarized in Section 16. A preliminary mine schedule is shown in Table 17.1. The LOM average head grades to the heap-leach operations are estimated to be

0.40g Au/t and 17.3g Ag/t. The expected LOM average grades for the milling operations are 0.61g Au/t and 40.91g Ag/t.

The mine schedule indicates that the heap leach will operate throughout the mine life, receiving ore initially from Florida Mountain, lasting though Year 7. DeLamar ore deliveries will start in Year 2 and continue through Year 17. The mill will start receiving non-oxide ore from both Florida Mountain and DeLamar in Year 3 and will continue until Year 17.

17.2 Process Design

The flowsheets developed for the DeLamar project PFS are based on the metallurgical testing and interpretation presented in Section 13. The following subsections provide a summary of the main components of the process design criteria, a description of the PFS process flowsheets, the major process equipment selected for the project, the primary buildings required to support the major process equipment, a description of the primary process support infrastructure including the water systems, power, process air systems, and the tailing handling system.

The primary crushing system mass flow rates are based on 75% plant availability and the combined required capacity to maintain both oxide and non-oxide stockpile levels. The oxide secondary and tertiary crushing system mass flow rates are based on 80% plant availability through ore stacking on the pad. The non-oxide ore mass flow rates are based on a plant availability of 92%. For simplicity, the estimated availability used in sizing each unit operation is a combination of mechanical availability and equipment utilization and, therefore, reflects estimated operating times. Every operation may have its own definitions that may differ according to its needs.

Table 17-1 Heap Leach and Mill Feed Schedules

Year	Heap-leach Ore									Mill Ore								
	Florida Mountain			DeLamar			Total			Florida Mountain			DeLamar			Total		
	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag
PreProd	610	0.42	10.55				610	0.42	10.55									
1	10,287	0.50	13.43				10,287	0.50	13.43									
2	10,596	0.46	10.44	1,953	0.44	20.85	12,548	0.46	12.06									
3	9,474	0.37	8.16	3,161	0.45	30.08	12,635	0.39	13.64	1078	1.09	15.57	904	0.54	77.47	1,982	0.84	43.80
4	10,993	0.42	11.76	79	0.30	12.10	11,072	0.42	11.77	1885	0.77	19.35	275	0.47	58.17	2,160	0.73	24.30
5	3,386	0.39	16.20	9,214	0.35	12.71	12,600	0.36	13.65	1863	0.67	19.56	297	0.62	53.40	2,160	0.67	24.21
6	15	0.33	24.54	12,585	0.38	18.50	12,600	0.38	18.51	926	0.38	8.00	1,234	0.89	28.74	2,160	0.67	19.85
7	348	0.16	11.22	10,566	0.32	21.91	10,914	0.31	21.57	558	0.28	6.74	1,608	0.54	47.40	2,166	0.48	36.94
8				6,325	0.29	33.16	6,325	0.29	33.16	445	0.28	6.74	1,715	0.46	54.19	2,160	0.43	44.41
9				997	0.19	44.62	997	0.19	44.62	492	0.28	6.74	1,668	0.52	47.50	2,160	0.47	38.21
10				296	0.46	24.02	296	0.46	24.02	740	0.28	6.74	1,420	0.43	42.39	2,160	0.38	30.18
11				450	0.60	57.84	450	0.60	57.84	346	0.28	6.74	1,819	0.60	63.44	2,166	0.55	54.37
12				259	0.70	77.92	259	0.70	77.92	379	0.28	6.74	1,781	0.74	61.76	2,160	0.66	52.11
13				176	0.79	77.57	176	0.79	77.57	300	0.28	6.74	1,860	0.78	67.77	2,160	0.72	59.28
14				170	0.74	78.07	170	0.74	78.07	307	0.28	6.74	1,853	0.86	66.58	2,160	0.78	58.07
15				200	0.61	74.82	200	0.61	74.82	277	0.28	6.74	1,889	0.78	60.26	2,166	0.71	53.41
16				103	0.48	71.64	103	0.48	71.64	302	0.28	6.74	1,858	0.68	55.00	2,160	0.62	48.25
17				0.008	0.78	143.5	0.008	0.78	143.5	656	0.28	6.74	506	0.43	23.45	1,162	0.35	14.01
Total or Average	**45,708**	**0.43**	**11.40**	**46,533**	**0.36**	**23.10**	**92,241**	**0.40**	**17.30**	**10,555**	**0.53**	**12.26**	**20,688**	**0.65**	**55.52**	**31,243**	**0.61**	**40.91**

17.3 Heap Leach Operation

The proposed initial processing facility is designed to process the Florida Mountain and DeLamar oxide and mixed ore types using conventional heap leaching for the extraction of precious metals and solution recovery using Merrill-Crowe zinc cementation process. Table 17.2 lists the major design criteria for the heap-leach process. The proposed process is depicted in the simplified flow sheet shown in Figure 17.1. The arrangement of the heap-leach facilities is shown in Figure 17.2.

Table 17-2 Heap Leach Major Design Criteria

Parameter	Florida Mountain	DeLamar
Processing Scheme	Crush/Heap Leach	Crush/Agglomerate/Heap Leach
Crushing Circuit Configuration	Three Stage Crushing	Three Stage Crushing with Drum Agglomeration
Crush Size	80% -12.7 mm	80% -12.7 mm
Heap Stacking Method	Overland Conveyor with Radial Stacker	Overland Conveyor with Radial Stacker
Leach Cycle, days	120	120
Lift Height, m	10	10
Solution Application Rate, L/hr/m^2	6.1	6.1
Nominal Barren Solution Flow, m^3/hr	2,119	2,119
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	2,040	2,040

17.3.1 Heap-Leach Crushing Plant and Agglomeration

The DeLamar heap-leach crushing plant will comprise three stages of crushing, starting with a gyratory crusher for primary crushing, followed by a standard cone crusher for secondary crushing, and finally, two short-head cone crushers for tertiary crushing. Table 17.3 is a list of the major equipment in the crushing plant.

Figure 17-1 Simplified Flow Sheet of the DeLamar Project Heap-Leach Facility

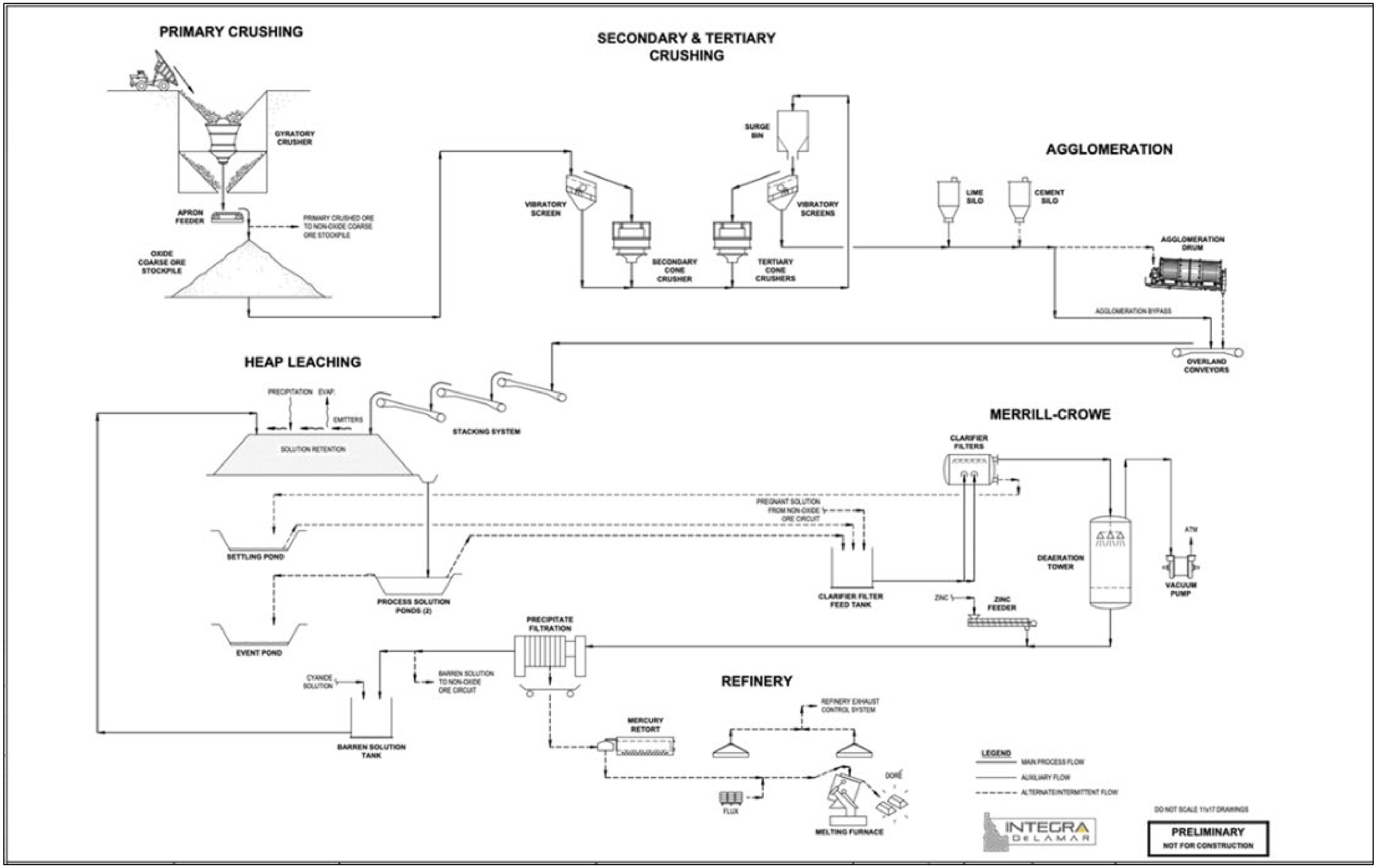


Figure 17-2 General Layout of the DeLamar Project Process Facilities

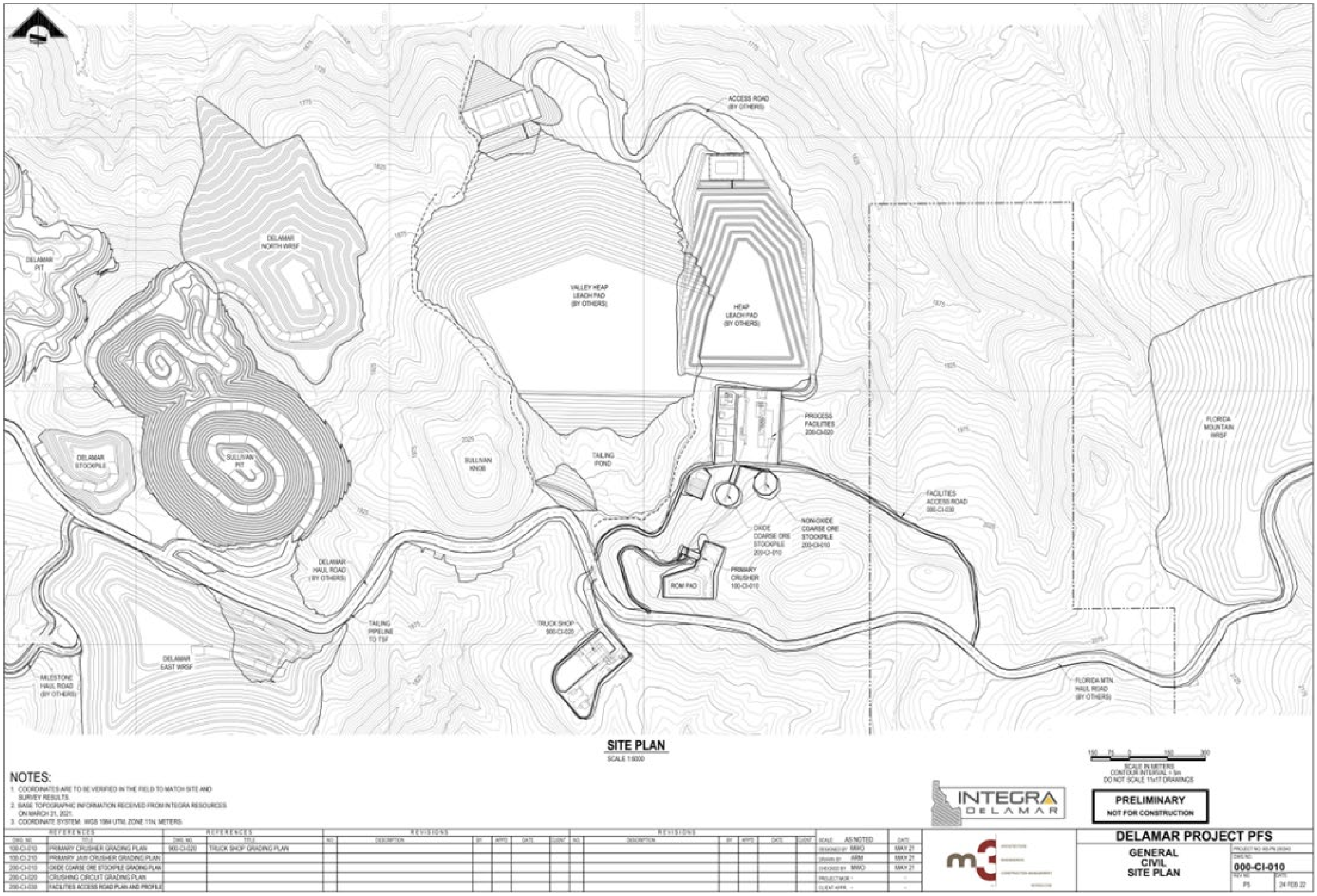


Table 17-3 List of Main Mechanical Equipment for the Heap-Leach Crushing Plant

Equipment	Number	Description	Installed kW
Primary Crusher	1	Gyratory Crusher, 42" x 65"; 152 mm (6-inch) OSS*	448
Secondary Crusher	1	Standard Head Cone Crusher; Metso MP1250 (o/e)*; CSS* = 35 mm (1-3/8-inch)	933
Tertiary Crusher	2	Short Head Cone Crusher; Metso MP1250 (o/e); CSS = 19 mm (3/4-inch)	933 (each)
Secondary Screen	1	Inclined, Vibrating, Double Deck; 1,823 dMTPH (Flow Sheet); 3 m x 7.3 m) (10 ft x 24 ft) (est); Apertures: 76.2 mm (3 inches), 31.75 mm (1.25-inch)	112
Tertiary Screens	2	Inclined, Vibrating, Single Deck; 2,349 dMTPH (Flow Sheet); 3.66 m x 7.31 m (12 ft x 24ft) (est); Apertures: 21 mm (0.82-inch)	112 (each)
Primary Crusher Discharge Conveyor	1	2,278 dMTPH, 152 cm (60-inch) wide, 72.2 m (237 ft) long, 5.2 m (17 ft) lift	224
Coarse-Ore Stockpile Feed Conveyor	1	2,278 dMTPH, 152 cm (60-inch) wide, 285 m (934 ft) long, 24.4 m (80 ft) lift	746
Secondary Screen Feed Conveyor	1	1,823 dMTPH, 122 cm (48-inch) wide, 191.4 m (628 ft) long, 15.2 m (50 ft) lift	373
Transfer Conveyors	2	4,698 dMTPH, 122 cm (48-inch) wide, 192 m (630 ft) long, 3 m (10 ft lift; and 169 m (555 ft) long, 24.4 m(80 ft) lift	298 & 933
Crushing Circuit Product Transfer Conveyor	1	1,823 dMTPH, 122 cm (48-inch) wide, 247 m (810 ft) long, 6 m (20 ft) lift	261
Agglomeration Feed Conveyor	1	Batch Use, 1,823 dMTPH, 122 cm (48-inch) wide	75
Agglomerated Product Conveyor	1	Batch Use, 1,823 dMTPH, 122 cm (48-inch) wide	75
Agglomeration Drum	1	1,823 dMTPH; 3.66 m Dia x 12.2 m Length, Tire Driven; w/ VFD	261

**OSS = open-side setting; CSS = closed-side setting; o/e = or equivalent*

A specialized ROM conveyance system ("Railveyor") will primarily deliver ROM to the primary crusher dump pocket. The crushed product will drop to the primary crusher product surge bin, which is equipped with an apron feeder that will deliver the ore to the primary discharge conveyor and on to a diverter cart. The diverter cart will allow primary crushed oxide and mixed ore to be routed to the oxide coarse-ore stockpile via a dedicated oxide ore stockpile feed conveyor, or non-oxide ore to be routed to the non-oxide ore stockpile via a separate, non-oxide ore stockpile feed conveyor.

Three apron feeders, two operating and one standby, will reclaim ore from the heap-leach coarse-ore stockpile and load it to the secondary screen feed conveyor. The secondary screen will separate +32-millimeter (1.25-inch) material and feed it to the secondary crusher. The screen undersize drops to a transfer conveyor and joins the secondary crusher product to the tertiary crusher feed bin.

From the tertiary crusher feed bin, the material is split between two tertiary screens with an aperture of 21 millimeters (0.82-inch). The oversize is fed to the tertiary crushers, whose products are recycled back to the tertiary crusher feed bin and on to the tertiary screens. The undersize of the tertiary screen is the final product of the crushing plant and has an estimated size of 80% finer than 12.7 millimeters (0.5-inch).

The crushing plant product is discharged to the crushing circuit product transfer conveyor where lime is added from a lime silo. The ore is then transferred to the stacking system feed conveyor or, alternatively, to the agglomeration feed conveyor, via a diverter cart.

Ore agglomeration will be achieved mainly at the transfer points of the grasshopper conveying system. Moisture will be supplied to the process by adding fresh water to the ore on the stacking system feed conveyor. For about 45% of the DeLamar oxide ore, agglomeration will be performed in an agglomeration drum using raw water or barren solution and cement, which will be added to the ore on the crushing circuit product transfer conveyor.

17.3.2 Stacking and Heap Leaching

The final crusher or agglomerator product reports to the stacking system feed conveyor that feeds the first of the grasshopper conveyors. The initial stacking system will comprise 20 grasshopper conveyors, and an index feed conveyor and a horizontal index feed conveyor that feeds the heap-leach stacker conveyor. The initial conveying and stacking equipment are listed in Table 17.4.

Table 17-4 List of Conveying and Stacking Equipment for the Heap-Leach Pad

Equipment	Number	Description	Drive, kW
Stacking System Feed Conveyor	1	1,823 dMTPH	298
Grasshopper Conveyors	20	1,823 dMTPH	112
Index Feed Conveyor	1	1,823 dMTPH	149
Horizontal Index Conveyor	1	1,823 dMTPH	112 (each)
Heap Leach Stacker Conveyor	1	1,823 dMTPH	298

The heap-leach pad will be a dedicated pad, which will be built in stages as mining progresses. The heap-leaching facility will consist of two dedicated heap-leach pads, namely the Jacob's Ridge leach pad and the Valley leach pad. Each leach pad will have its own pregnant leach solution ("PLS") pond. The PLS pond at the Jacob's Ridge leach pad will be sized to contain the operating volume, the drain down volume, and a 100-year 24-hour precipitation event, plus freeboard.

After stacking, pipe headers 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 6.10 L/hr/m^2 for the preliminary leach cycle of 120 days. The cyanide solution, applied at a nominal flow rate of 2,119 cubic meters per hour (9,329.7 gpm) to the heap surface, will percolate though the heap until it reaches the impervious leach pad liner at the bottom of the heap. The PLS will flow by gravity to the collection point of the leach pad and on to the pregnant solution pond for each leach pad. From the pregnant ponds, the PLS will be pumped to the clarifier filter feed tank in the Merrill-Crowe plant. The heap-leach solution application rate may be adjusted during the leach cycle depending on the available area for irrigation.

Heap-leach modeling performed by Integra's metallurgical consultant, Mr. Michael Botz of Elbow Creek Engineering Inc. in Billings, Montana indicates the upper-end sustained flow rates of barren solution and pregnant solution will be approximately 1,950 and 1,810 cubic meters per hour (8,585.6 gpm to 7,969.2

gpm), respectively. These flow rates are predicted for the period when the heaps are stacked with ore at about 35,000 t/d. The difference of 140 cubic meters per hour (616 gpm) between the flows arises from evaporation plus uptake of solution by the ore for initial wetting. The design of the Merrill-Crowe metal recovery circuit must accommodate the predicted pregnant solution flow rate of 1,810 cubic meters per hour (7,969.2 gpm). With an added 10% design margin, the hydraulic sizing basis for the Merrill-Crowe circuit would be approximately 2,000 cubic meters per hour (8,805.7gpm).

For the case where a tank leaching circuit is operated at the site along with the heap leach, an additional 100 cubic meters per hour (440 gpm) of pregnant solution would also be routed to the Merrill-Crowe circuit. This would give a total predicted feed flow rate to the Merrill-Crowe circuit of 1,910 cubic meters per hour (8,409.5 gpm). With an added 10% design margin, the hydraulic sizing basis for the Merrill-Crowe circuit would be 2,100 cubic meters per hour (9,250gpm). M3 Engineering has selected a flow rate of 2,119 cubic meters per hour (9,330 gpm) for the purposes of the PFS.

The design of the leach pad and solution ponds are discussed in more detail in Section 18.0.

17.3.3 Heap-Leach Solution Pond Operation

There are three solution ponds, and all will be constructed in Phase 1. One of the solution ponds will be located at the north end of the Jacob's Ridge heap and two ponds will be located at the north (lower) end of the valley heap. The pond at Jacobs Ridge is sized to contain an operating volume of 15,140 cubic meters (4,000,000 gallons) of pregnant solution storage, plus 100% of a 100-year, 24-hour precipitation event on the ridge portion of the heap, a volume of 24,000 cubic meters (6,342,400 gallons), plus a drain-down volume of 24,224 cubic meters (6,400,000 gallons), plus the volume of 0.6 meters (2.0 feet) of freeboard, 7,066 cubic meters (1,867,000 gallons). As dimensioned, the overall volume of the Jacobs Ridge pond is 71,642 cubic meters (18,928,000 gallons). This pond is designed to never overflow. Following a large storm event, the pond should be pumped at a minimum of 568 liters per second (9,000 gpm) and if there is excess pumping capacity over that needed for application to the heap, the extra flow should be routed to the concentrate leach TSF for temporary storage.

The two ponds at the lower (north) end of the valley are an operating pond and an event pond. The operating pond will be double lined with leak detection and will contain 44,920 cubic meters (11,868,000 gallons) of pregnant solution storage as an operating volume, plus 0.6 meters of freeboard. Excess flows from a precipitation event up to a 100-year, 24-hour event will flow through a spillway from the operating pond to a single-lined event pond where they will be temporarily stored until they can be pumped into the operating pond as makeup pregnant solution for return to the plant. The capacity of the event pond will be 71,130 cubic meters (18,792,659 gallons).

Pregnant solutions will be pumped to the process area in a welded steel pipeline following the access road from the valley pond area, past the Jacobs Ridge pond area to the plant for extraction of precious metals.

17.3.4 Heap-Leach Production Forecasting

Gold and silver production forecasts for the Jacob's Ridge and Valley heap-leach pads were provided by Elbow Creek Engineering. The forecasts were developed using a heap-leach production model that follows the modeling technique described by Botz and Marsden (2019). The dynamic model provides monthly estimates of gold and silver productions according to the planned construction and operation of the heaps. The forecasts take into account kinetic leach reactions, transport of leached metal to the heap liner, and holdup of leached metal within the heap pore moisture inventory. Kinetic leach rates were sourced from column testwork performed by McClelland for each ore type. Ultimate (final) extractions for gold and silver by ore type were also provided by McClelland. Scale-up discount factors were applied to the extractions by McClelland to account for full-scale leach inefficiencies, plus the small residual of leached metal that will be retained in the heap pore moisture and the end of operations.

A summary of the gold and silver forecasts for the heap-leach pads is provided in Table 17.5. The total forecast LOM gold recovery is estimated to be 843,800 oz, and the total forecast LOM silver recovery is estimated to be 19,168,000 oz.

With any heap-leach production forecast model, there will always be a variance between modeled and actual metal productions. This variance arises from inherent variabilities associated with many aspects of heap operation, including variability in ultimate extractions, leach times, lift heights, irrigation practices, mineralogy, particle size distributions, permeability, etc. The variances that could generally be expected for a well-understood heap are shown in Table 17.6 (Marsden and Botz 2017).

Table 17-5 Heap-leach Gold and Silver Production Forecasts

Year	Gold [oz]		Silver [oz]	
	Year	Cumulative	Year	Cumulative
1	9,900	9,900	152,000	152,000
2	119,400	129,300	1,956,000	2,108,000
3	138,700	268,000	1,863,000	3,971,000
4	111,600	379,600	1,715,000	5,686,000
5	107,700	487,300	1,867,000	7,553,000
6	103,200	590,500	1,838,000	9,391,000
7	101,300	691,800	2,184,000	11,575,000
8	78,300	770,100	2,606,000	14,181,000
9	42,500	812,600	2,336,000	16,517,000
10	10,400	823,000	1,285,000	17,802,000
11	2,700	825,700	105,000	17,907,000
12	5,800	831,500	333,000	18,240,000
13	3,300	834,800	254,000	18,494,000
14	2,900	837,700	187,000	18,681,000
15	2,500	840,200	171,000	18,852,000
16	2,300	842,500	188,000	19,040,000
17	1,200	843,700	110,000	19,150,000
18	100	843,800	11,000	19,161,000
19	0	843,800	4,000	19,165,000

Year	Gold [oz]		Silver [oz]	
	Year	Cumulative	Year	Cumulative
20	0	843,800	3,000	19,168,000
Total	843,800		19,168,000	

Table 17-6 Typical Achievable Variances Between Modeled and Actual Metal Productions

Forecast Period	1σ Level	2σ Level
Monthly	±20%	±40%
Quarterly	±15%	±30%
Annually	±6%	±12%
Life of Heap	±1%	±2%

17.4 Milling Operations

The design of the mill for non-oxide ore includes a primary crusher, which is shared with the oxide ore crushing plant, a grinding circuit, flotation, very fine grinding of concentrate, and a cyanidation leach/CCD circuit for the recovery of precious metals. Table 17.7 is a list of the main design criteria for the mill. The proposed simplified process flow sheet for the mill is shown in Figure 17.3. A list of major equipment in the mill is given in Table 17.8.

Table 17-7 Main Design Criteria for Concentrator and Cyanidation Plants

Parameter	Value
Milling – Non-Oxide Material Type	
Overall Milling Scheme	Comminution/Flotation/Conc. Leach
Comminution Circuit Configuration	Primary Crusher, SAG Mill and Ball Mill
Crush Size	80% -120 mm
Primary Grind Size	80% -150 µm
Flotation Cell Type	Tank Cell
Flotation Circuit Configuration	Rougher-Rougher Scavenger
Rougher Flotation Retention Time, min	25
Rougher Scavenger Retention Time, min	20
Flotation Tailing Handling Method	Thicken, Tailing Storage Facility
Concentrate Mass (for regrind) Percent of Total Feed	10%
Concentrate Leach Solids Density, % wt	40
Concentrate Leach Retention Time	24 hrs.
Regrind Size	80% -20 µm
Au Recovery	
Florida Mountain	83%

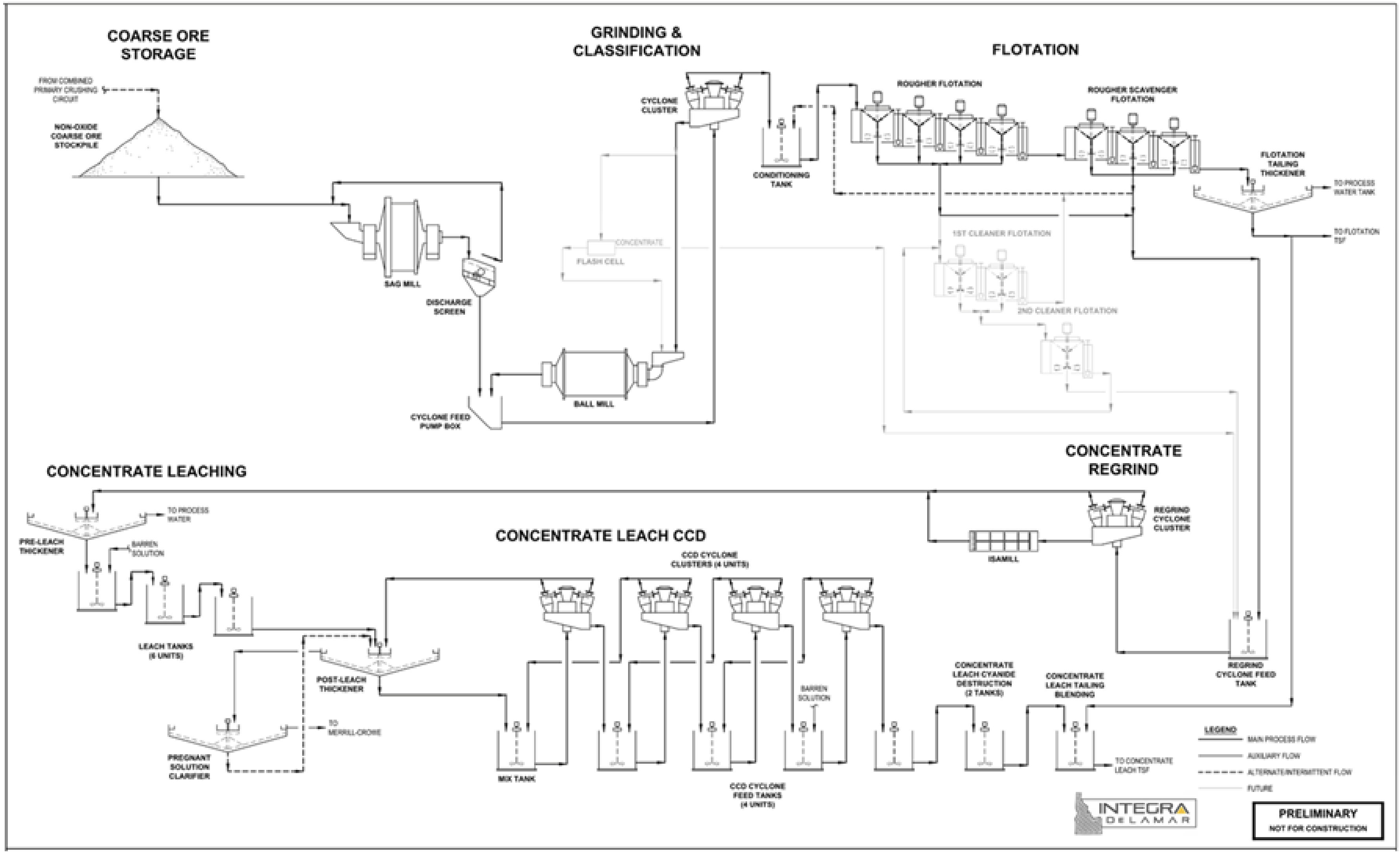
Parameter	Value
Milling – Non-Oxide Material Type	
DeLamar	37%
Ag Recovery	
Florida Mountain	72%
DeLamar	75%
Leach Tailing Handling Method	Concentrate Leach Tailing Storage Facility

Table 17-8 List of Main Mill Equipment

Equipment	Number	Description	Drive, kW
Primary Crusher	1	Uses primary crusher at Oxide Ore Processing	448
SAG Mill Feed Conveyor	1	340 dMTPH; 169 m (553 ft) L, 4.3 m (14 ft) Drop	112
Semi-Autogenous (SAG) Mill	1	6.7 m diam x 4.57 m EGL* (22 ft x 15 ft), with internal grates & pulp discharge	2,984
SAG Mill Discharge Screen	1	Inclined, Vibrating, Single Deck; 340 dMTPH (Flow Sheet); 3 m x 6 m (10 ft x 20 ft) (est); Apertures: 8 mm	75
Ball Mill	1	4.57 m diam x 9.14 m EGL (15 ft x 30 ft)	2,984
Primary Hydrocyclone Cluster	1	1,697 m^3/h (7,472 gpm); 10-place Cluster (est); 20-inch Cyclones; 5-Operating; 3-Stand-by; 2-Spare Ports	n/a
Rougher Flotation	4	100 m^3 Tank Cell (est)	112 (each)
Rougher Scavenger Flotation	3	100 m^3 Tank Cell (est)	112 (each)
Utlra-fine Regrind Mill	1	ISAMill Model M5000 (o/e)	1,119
Regrind Cyclone Cluster	1	91.4 m^3/h (402 gpm); 10-place Cluster (est); 6-inch Cyclones; 8-Installed, 2 Spare Ports	n/a
Pre-Leach Thickener	1	12.2 m (40 ft) Diameter; High Rate	15
Leach Tanks	7	218 gpm; 24-hr Circuit Retention; 20ft D x 24ft H; 54k gal Total per Tank; Open Top; rubber-lined carbon steel	19 (each)
Post-Leach Thickener	1	12.2 m (40 ft) Diameter; High Rate	15
Pregnant Solution Clarifier	1	12.2 m (40 ft) Diameter; High Rate	15
CCD Cyclone Clusters	4	107 – 158 m^3/h (470 – 697 gpm); 10-place Cluster (est); 6-inch Cyclones; 8-Installed, 2 Spare Ports	n/a
CCD Mix Tanks	4	158 m^3/h (697 gpm); 5-min Retention; 3 m dia x 3.66 m H (10 ft x 12 ft); Open Top; rubber-lined carbon steel	3.7 (each)
Cyanide Destruction Tank	2	42 m^3/h (185 gpm); 60-min Retention; 3.66 m D x 4.88 m H (12 ft x 16 ft; Open Top; rubber-lined carbon steel	19 (each)

EGL is effective grinding length.

Figure 17-3 Simplified Process Flow Diagram of the Concentrator and Cyanide Leach Plants



17.4.1 Comminution

Primary crushing of non-oxide ore will be performed in the same gyratory crusher as the heap-leach operation. Coarse ore will be retrieved and sent to a separate non-oxide coarse ore stockpile, from which ore is reclaimed by two reclaim feeders, and transferred to the SAG mill feed conveyor.

The grinding circuit comprises a SAG mill and a ball mill. The SAG mill will operate in a closed circuit with a vibrating screen. The oversize of the screen will be conveyed back to the SAG mill via the screen oversize conveyors and SAG mill feed conveyor. The undersize slurry will flow into the ball mill cyclone feed pump box as fresh feed to the ball mill circuit.

The ball mill will operate in a closed circuit with the primary hydrocyclone cluster, which sends the underflow back to the ball mill and the overflow to the flotation circuit via the trash screen. The target grind for the ball mill circuit is 80% finer than 150 microns.

Possible future additions to the circuit include a pebble crusher and flash flotation.

17.4.2 Flotation and Regrind

Ground ore will first be conditioned with potassium amyl xanthate ("PAX"), ethyl sec-butyl dithiophosphate ("Aerofloat 208") and frother ("MIBC") in an agitated tank for 15 minutes at 30% solids. Sodium metasilicate may also be added as a dispersant with high-clay ores. Once conditioned, the slurry flows by gravity to the rougher flotation (four cells) and rougher scavenger flotation (three cells) banks. Additional flotation reagents will be introduced to the slurry between the rougher and rougher scavenger banks. The combined rougher and rougher scavenger residence time is 45 minutes, to produce a combined flotation concentrate at an estimated mass pull of 10% for DeLamar ores and 5% for Florida Mountain ores.

Other reagents are added to the flotation process depending on the ore being processed. These are included in Table 17.9

Future plans for the flotation plant could include two stages of cleaning to further upgrade the concentrate prior to regrinding.

The rougher scavenger flotation tailing will flow by gravity to the flotation thickener to be dewatered to 55% solids by weight. The thickener overflow is to be pumped to the process water tank. The thickened underflow is to be pumped to the tailing storage facility. A portion of the rougher flotation tailing will be blended with concentrate leach tailing at a 1:1 solid mass ratio to improve the settling characteristics of the finer tailing.

The combined flotation concentrate will proceed to the regrind circuit, which consists of an ISAMill and a regrind cyclone cluster. The concentrate will enter the circuit through the regrind cyclone feed tank and will then be pumped to the regrind cyclone cluster. The underflow of the regrind cyclones will be fed by gravity to the ISAMill, which operates in open circuit. The overflow of the regrind cyclones and the product of the ISAMill both report to the pre-leach thickener, which will thicken the leach feed to 50%

solids. The target grind of the leach feed is 80% finer than 20 microns. The pre-leach thickener overflow will be pumped to the process water tank.

17.4.3 Concentrate Leaching

The concentrate leaching circuit comprises six leach tanks operating in series followed by a CCD system. The leach tanks are designed to provide a residence time of four hours each, for a total leach time of 24 hours. Milk-of-lime ("MOL") is added to slurry at the pre-leach thickener feed while sodium cyanide is added to the leach feed splitter box with stage addition of either into the leach tanks as required. Air is introduced below the agitators of each leach tank, supplied by one leach air blower.

The final leach residue will be pumped to the CCD circuit, which starts with a post-leach thickener and a series of four CCD cyclone clusters, which are supported by four agitated CCD mix tanks and four pumps (one operating per cluster). The cyclone clusters in series serve the same purpose as thickeners in series, with the post-leach thickener serving in the position of the first CCD stage. Overflow from the post-leach thickener is clarified in a thickener clarifier before being pumped to the Merrill-Crowe plant.

The underflow of the final CCD cyclone cluster will be pumped to the CCD cyanide destruction plant to destroy cyanide with metabisulfite and air. Once detoxified, the concentrate leach tailing is deposited in a separate concentrate leach tailing storage facility. In the future, concentrate leach tailing may be mixed with the fresh heap-leach feed in an agglomeration drum to extend its leach time and reduce the need for tailing detoxification and separate tailing storage facility.

17.5 Merrill-Crowe Plant and Refinery

The PLS reporting to the pregnant solution pond and, starting year 3, solution from the CCD circuit pregnant solution clarifier will be pumped to the clarifier filter feed tank at the Merrill-Crowe plant. Solution clarification will be performed by three clarifying filters arranged to operate in parallel, with two operating and one on standby. The filters will have been precoated with diatomaceous earth ("DE") to aid filtration. The solution may be infused with DE as body feed when required. The clarified solution then proceeds to the deaeration tower where it will be introduced into an evacuated chamber to remove as much dissolved oxygen as possible. After deaeration, powdered zinc, cyanide, and lead nitrate will be added to the solution to initiate an exchange redox reaction where zinc metal loses electrons to gold and silver, thereby reducing gold and silver to their metallic state and oxidizing zinc to form cyano complexes in solution.

The mixture will then be pumped to three recessed plate and frame filters operating in parallel, two operating and one standby. Precipitation of gold and silver by the exchange reaction continues as the solution makes its way to the Merrill-Crowe precipitate filters and reaches completion inside the filters. All the precipitated gold and silver will remain in the filter presses until they are discharged when the filters are full. The filtrate solutions, stripped of values, will report to the barren solution tank. Additional cyanide and caustic may be introduced into the barren solution tank before it is recycled to the heap and to the leaching tanks and CCD circuit when the mill is operating.

Gold and silver precipitates collected by the filter presses will be dried in a retort to remove moisture and mercury before they are fluxed and smelted in an induction melting furnace. At the end of smelting, molten metal is poured into bullion molds to produce the final plant product, doré bars, which are packed for shipment.

The slag, which is poured first into a slag pot, will be weighed, sampled, and stored for further processing if required, or transferred to the heap or fed to the ball mill when the mill is operating.

Off gases from the retort furnace will be treated to remove mercury by condensation. The remaining gas is subsequently passed through a bed of sulfur-impregnated carbon. Off gases from the induction furnace will be filtered, scrubbed with water sprays, and finally passed through a bed of sulfur-impregnated carbon.

A settling pond will receive backwash water from the clarifier filters to settle diatomaceous shale for solution recycling. It will also receive water from the refinery wet scrubber.

17.6 Reagents

The main reagents to be used in leach processes in the heap-leach pad and the mill will be lime and sodium cyanide. In addition, the mill will use a suite of flotation reagents consisting of two flotation collectors, a frother, and two other conditioning reagents as required. The Merrill-Crowe process will use reagents that are standard in typical Merrill-Crowe operations. Table 17.9 is a summary of the reagents projected to be used in this project including points of addition and estimated consumption rates.

The MOL system consists of a lime slaker and MOL distribution tank. Pebble lime will be stored in a silo and metered into a vertical grinding mill by a screw feeder. The MOL produced will be pumped to a 4.3-meter (14-foot) diameter by 4.9-meter (16-foot) high MOL storage/distribution tank. MOL will be pumped to the solution regeneration tanks through a MOL loop.

Pebble lime will also be added to the oxide and mixed ores from another lime silo to the crusher product transfer conveyor.

17.7 Tailing Storage Facilities (TSF) and Water Consumption

As with most operating mines, as much water as possible is planned to be recycled from slurry thickeners and mill tailing. Water lost to evaporation, heap-leach ore, and mill tailing needs to be replenished by raw water. In addition, raw water is used for dust suppression on mine roads and potentially for potable water.

17.7.1 Tailing Storage Facilities and Water Reclamation

The plans for tailing storage for the mill operations include two facilities – one to store flotation tailing, and the other, a smaller facility, to store concentrate leach tailing.

Water reclaimed from the flotation TSF will report to the process water tank, to combine with the tailing thickener overflow, pre-leach thickener overflow, and make-up raw water coming from the oxide circuit freshwater system. The process water tank will supply water to the grinding circuit, rougher flotation, flocculant make-down, and regrind circuits.

The smaller tailing storage facility will be dedicated to concentrate leached tailing to isolate the stream within the cyanidation loop. Water reclaimed from this TSF will be pumped to the CCD circuit to supply part of the wash water for CCD.

17.8 Water Consumption

The DeLamar project is projected to require a total average of 249.6 cubic meters per hour (1,099 gpm) of raw water makeup to sustain the operation, for a total yearly requirement of 2.2 million cubic meters (581 million gallons). The water usage is broken down as follows:

- Heap-leach operation 103.7 cubic meters per hour (457 gpm);
- Milling operation 130.2 cubic meters per hour (573 gpm); and
- Mine dust suppression 15.7 cubic meters per hour (69 gpm).

The actual requirement will fluctuate according to the season. A discussion of the site-wide water balance is presented in Section 18.8.

Table 17-9 Main Process Reagents and Consumables

Reagent	Area or Point of Addition	Dosage	Dosage unit
Sodium Cyanide	Leach, Barren Solution Pond & Merrill-Crowe	0.3 to 0.6	kg/t
	Mill – Cyanidation	0.26 to 1.25	kg/t
Caustic Soda	Sodium Cyanide Distribution Tank	0.005 to 0.025	kg/t
Lime	Heap Leach	0.3 to 2.4	kg/t
	Mill	0.1 to 0.7	kg/t
	Cyanide Destruction	0.6 to 1.2	kg/t conc
Cement	Heap Leap – Agglomeration	2.8*	kg/t
Zn Dust, estimate	Merrill-Crowe	25.8	kg/kOz Au
		47.1	kg/kOz Ag
Lead Nitrate, estimate	Merrill-Crowe	15	ppm in PLS
Diatomaceous Earth (DE), estimate	Merrill-Crowe	45.4	kg/filter batch
Melting Flux, estimate	Refining	5.5	g/oz of metal
Flocculant, estimate	Tailing Thickener, Pre-Leach Thickener, & Post-Leach Thickener	20 to 30	g/t solids
Antiscalant, estimate	Process Water Pump, Barren Solution Pump	5	g/t of solution
Potassium Amyl Xanthate (PAX)	Flotation	0.025	kg/t
Ethyl sec-butyl dithiophosphate (Aerofloat 208)	Flotation	0.05	Kg/t
Methyl Isobutyl Carbinol (MIBC) Frother)	Flotation	0.1	kg/t

Reagent	Area or Point of Addition	Dosage	Dosage unit
Sodium Carbonate	Flotation	0.25**	kg/t
Sodium Silicate (for high-clay ores)	Flotation	0.025***	kg/t
Sodium Metabisulfite	Cyanide Destruction	1 to 2	kg/t conc
Copper Sulfate	Flotation/Cyanide Destruction	0.016	kg/t conc

**Assumes 45% of DeLamar ore requires agglomeration.*
***Factored (assuming required for 25% of ore)*
****Factored (assuming required for 10% of ore)*

17.9 Blowers and Compressors

The design of the leach pad and mill facilities includes blowers and compressors to provide general plant air, aeration for flotation, leach, detoxification, dust control, and instrument air. A list of air equipment is given in Table 17.10.

Table 17-10 List of Blowers and Compressors for Supply Plant and Instrument Air

Equipment Tag	Service Area	Power, kW
200-CM-003	Air Compressor for Fine Crushing and Dust Collection	56
500-DC-014-BL	Refinery Wet Scrubber Blower	75
900-CM-004	Merrill-Crowe Plant Air Compressor	56
320-CM-003	Grinding Area Compressed Plant Air Supply and Dust Collection	56
420-BL-003	Rougher Flotation Blower	261
420-BL-005, -006	Cleaner Flotation Blower (Future); 1 operating, 1 standby	149 (ea)
420-CM-004	Cleaner Flotation Area Compressed Plant Air (Future)	37
440-BL-001	Leach Air Blower, 1360 Nm^3/h (800 scfm), 138 kPa (20 psig)	75
460-BL-009	CCD Cyanide Destruction Blower, 1020 Nm^3/h (600 scfm), 97 kPa (14 psig)	30
820-BL-005	Lime Unloading Blower, batch use	11

17.10 Power Consumption

The total connected power load is estimated at 20,907 kW for the heap-leach operation and 12,473 kW for the mill. The connected power in each process area is given Table 17.11. The actual power drawn is calculated in the financial model.

Table 17-11 Summary of Connected Power for Heap-leach and Mill Operations

Area	Description	Connected Load, kW
HEAP-LEACH OPERATIONS		
100	Primary Crushing	1,607
200	Fine Crushing	5,798
250	Agglomeration	436
300	Conveying and Stacking	3,158
350	Heap Leaching	6,867
400	Merrill-Crowe	1,795
500	Refinery	368
650	Water Systems	795
800	Reagent Facility	25
900	Compressed Air	57
Total Heap Leach		**20,907**
NON-OXIDE ORE MILLING OPERATIONS		
120	Primary Crushing (Transfer Conveyors)	1,141
220	Crushed Ore Reclaim	204
320	Grinding	7,094
420	Flotation and Regrind	2,309
440	Concentrate Leaching	259
460	Concentrate CCD	419
620	Flotation Tailing Thickening and Handling	498
670	Water Systems	297
820	Reagents	252
Total Mill		**12,473**

17.11 Control Systems

A crusher control room in the primary crusher area will be the operating and control center for the crushing plants, particularly for the primary crusher and Railveyor system. A central control room ("CCR") will be located inside the Merrill-Crowe Plant building. From the CCR control consoles, crushing, screening, material handling systems, reagents, pumping systems, aeration system, Merrill-Crowe plant, and utility systems will be monitored or controlled.

A computer room adjacent to the CCR will contain engineering workstations ("EWs"), a supervisory computer, historical trend system, management information systems ("MIS") server, programming terminal, network and communications equipment, and documentation printers. This will be primarily used for distributed control system ("DCS") development and support activities by plant and control systems engineers.

Although the facilities are normally controlled by the CCRs, local video display terminals are to be selectively provided on the plant floor for occasional monitoring and control of certain process areas. Any local control panels that are supplied by equipment vendors will be interfaced with the DCS for remote monitoring or control.

17.12 Assay and Metallurgical Laboratories

A laboratory building has been provided for in the capital cost estimate. Provision has been made for facilities that include sample receiving, sample drying, sample preparation, metallurgical laboratory, wet laboratory, and fire assay for mine and process plant samples.

17.13 Alternative Processing Options

Integra has considered several processing tradeoffs in terms of grind size, processing rates, etc., with and without the use of a high-pressure grinding roll ("HPGR") circuit. Also, the addition of flash flotation and cleaner flotation in the non-oxide plant are being considered for future installation. Finally, oxidation of flotation concentrates using the Albion process is being studied for inclusion in future technical reports.

17.13.1 High-Grade Heap Leaching Ore Processing

A tradeoff study was conducted to determine if high-grade heap-leach ore would justify a finer crush using three stages of crushing with agglomeration. The third-stage crusher would either be a cone crusher or an HPGR. In addition, the study also examined the viability of sending a portion of the high-grade heap-leach ore to the mill.

The parameters for the study are shown in Table 17.12 in the order of intensity of treatment. Table 17.12 includes the capital cost, operating cost, and metal recoveries for each option. For the tradeoff study, the capital and operating costs were based primarily on historical benchmarks and M3 Engineering's in-house database information. The costs in the tradeoffs are exclusively intended for comparison within the tradeoff.

Table 17-12 Processing Options for High-Grade Heap-Leach Ores

Option	HL mtpd	Mill mtpd	HL Crushing Circuit	Capex $M	Process	Ore	Florida Mountain			DeLamar		
							Au Rec, %	Ag Rec, %	Opex, $/t	Au Rec, %	Ag Rec, %	Opex, $/t
1	30,000	0	2-Stage Cone	$129.0	Heap Leach	Oxide	90%	40%	$2.55	85%	24%	$2.65
			50 mm			Trans	80%	30%	$2.85	66%	22%	$3.60
2	30,000	0	3-Stage Cone	$166.6	Heap Leach	Oxide	90%	65%	$2.90	85%	30%	$3.00
			Agglomerated 12.7 mm			Trans	85%	55%	$3.55	70%	30%	$4.00
3	30,000	0	3-Stage HPGR	$172.4	Heap Leach	Oxide	90%	69%	$3.20	85%	36%	$3.35
			Agglomerated 6.4 mm			Trans	85%	60%	$3.95	72%	34%	$4.40
4	25,000	5,000	2-Stage Cone	$266.6	Heap Leach	Oxide	90%	40%	$2.65	85%	24%	$2.75
			50 mm			Trans	80%	30%	$3.00	66%	22%	$3.70
					Mill	Oxide	93%	80%	$14.40	88%	67%	$14.70
						Trans	88%	89%	$25.40	75%	58%	$16.05
5	25,000	5,000	3-Stage Cone	$304.2	Heap Leach	Oxide	90%	65%	$3.00	85%	30%	$3.10
			Agglomerated			Trans	85%	55%	$3.65	70%	30%	$4.10
			12.7 mm		Mill	Oxide	93%	80%	$14.40	88%	67%	$14.70
						Trans	88%	80%	$15.40	75%	58%	$16.05
6	25,000	5,000	3-Stage HPGR	$307.6	Heap Leach	Oxide	90%	69%	$3.35	85%	36%	$3.45
			Agglomerated			Trans	85%	60%	$4.05	72%	34%	$4.55
			6.4 mm		Mill	Oxide	93%	80%	$14.40	88%	67%	$14.70
						Trans	88%	80%	$15.40	75%	58%	$16.05

17.13.2 Milling Options for DeLamar Non-oxide Ore

The base case scenario for milling non-oxide ore is to produce a flotation concentrate, fine grinding the concentrate and leaching the ground concentrate with cyanide. Options were studied to improve recoveries for DeLamar ores. These are shown in Table 17.13, which includes capital cost, operating cost, and metal recoveries for each option.

The first set of trade-off studies tested the base-case scenario at different mill throughputs, namely 5,000, 8,000 and 10,000 tonnes per day (Options 1 through 3). The second set of trade-off studies adds a leaching facility to leach flotation tailing (Options 4 and 5), at 5,000 and 8,000 tonnes per day. The third set of trade-off studies (Options 6 and 7) does not include a tailing leach facility but pre-oxidizes the concentrate using the Albion process at the finer grind of 10 microns.

Table 17-13 Milling Options

Option	Tonnage mtpd	Grind P80, m	Tailing Leach	Albion	Capex $M	Domain	DeLamar		
							Au Rec, %	Ag Rec, %	Opex, $/t
1	5,000	20	No	No	$135.0	Sullivan Gulch	45%	72%	$12.52
						Glen Silver	24%	64%	$12.52
2	8,000	20	No	No	$176.6	Sullivan Gulch	45%	72%	$11.30
						Glen Silver	24%	64%	$11.30
3	10,000	20	No	No	$203.2	Sullivan Gulch	45%	72%	$11.02
						Glen Silver	24%	64%	$11.02
4	5,000	20	Yes	No	$186.2	Sullivan Gulch	45%	75%	$17.96
						Glen Silver	29%	71%	$17.96
5	8,000	20	Yes	No	$250.7	Sullivan Gulch	45%	75%	$15.62
						Glen Silver	29%	71%	$15.62
6	5,000	10	No	Yes	$170.6	Sullivan Gulch	84%	84%	$20.06
						Glen Silver	74%	74%	$18.41
7	8,000	10	No	Yes	$226.4	Sullivan Gulch	84%	84%	$18.66
						Glen Silver	74%	74%	$17.01

Leaching of the flotation tail for DeLamar ores did not improve gold recovery and only slightly improved silver recovery. In contrast, pre-oxidation using the Albion process almost doubled gold recovery and significantly improved silver recovery.

17.13.3 Non-Oxide Ore Gravity Concentration

One option that was considered as a future installation is a gravity concentrator to process a bleed from the ball mill cyclone underflow. The gravity concentrate would be sent directly to concentrate regrind. The theory behind this option to is recover gravity-recoverable gold ("GRG") and silver to possibly increase recovery by catching gold and silver that could be missed by flotation.

17.13.4 Cleaner Flotation Stages

Cleaner flotation stages may be added in the future to improve concentrate grade and reduce the volume of concentrate that needs to be reground and leached. Tailing from the cleaner section will be returned to the rougher bank, which would increase overall flows because of the recycle streams. It is possible that concentrate grades, recoveries and mass pulls may be sufficient that the cleaner banks will not be necessary. The addition of cleaner flotation stages may be called for if the Albion Process would be included and the sulfide sulfur grade is not high enough (10% sulfide sulfide) to provide the required heat for the process.

17.13.5 Process Personnel and Staffing

Staffing requirements for process personnel have been estimated by M3 Engineering based on experience with similar-sized operations in the region. Total process personnel requirements are estimated at 41 persons for the heap-leach and Merrill-Crowe operation. An additional process personnel of 49 people would be required for the non-oxide mill circuit.

18.0 PROJECT INFRASTRUCTURE

The infrastructure for the DeLamar project has been developed to support mining and processing operations. This includes the access road to the facilities, power supply, Railveyor, communication, heap-leach pads, process plant, and ancillary buildings.

18.1 Access

The proposed site access road is similar in geometry and design to the existing site access road. The proposed typical road section consists of two 3.66-meter (12.0-foot) lanes with up to two 0.91-meter (3.0 feet) shoulders on either side and will be gravel surfaced. The site access road will have a 0.61-meter (2.0-foot) high safety berm on fill-slope sides and a roadside ditch and 1:1 or 2:1 (horizontal:vertical) cut slope for cut-slope sides. The roadside ditches will be periodically dewatered via corrugated metal culverts.

Haul road access between the DeLamar mine and Florida Mountain will need to be improved for use with the proposed mining equipment. This access will be utilized for delivery of all consumables, as well as any required construction materials and equipment. This will also be the primary access for all personnel working at Florida Mountain.

18.2 Heap-Leach Pad Construction

The heap-leach pads (“HLP” or “HLPs”) will be located immediately north of the crushing facility in portions of Sections 3, 4, 9 and 10, Township 5 South, Range 4 West. The site slopes northerly toward Jordan Creek at an average gradient of 12.5 percent. The HLPs will be constructed in two phases. The Phase 1 portion will be constructed on a feature locally identified as Jacobs Ridge and into an adjacent valley to the west (herein referred to as the “unnamed gulch” or the “valley”).

The Phase 1 pad area and solution ponds will be lined in accordance with the IDEQ Rules 58.01.13 – Rules for Ore Processing by Cyanidation. In accordance with the regulation, the lining system will consist of 0.6 meters (24-inches) of compacted clay overlain with an 80-mil thick high-density polyethylene (“HDPE”) liner – or approved equivalent.

The operating pond for the valley pad will be a double-lined pond with a capacity of 45,000 cubic meters (11.9 million gallons) and a single-lined event pond with a capacity of 71,160 cubic meters (18.8 million gallons). The operating pond will contain process solutions and will overflow into the event pond following large precipitation events. Additionally, two lined ponds are provided upstream of the valley portion of the HLP for short-term storage of storm runoff and excess process solution. Surface runoff will be diverted around the HLP with temporary ditches between Phase 1 and Phase 2 construction and permanent diversion ditches above Phase 2 construction.

Phase 2 portion of the HLP will consist of a westerly extension of the pad and tying in the area between the west side of the Jacobs Ridge pad and the east side of the Phase 1 valley pad. Construction of Phase 2 will begin two years ahead of when the extended pad is needed, assumed in year 3 of operation. Phase 2 construction will be performed in the same sequence of activities and will add approximately 30% to the pad footprint. The total volume of ore to be placed on the HLP is between 95 million tonnes and 100 million tonnes which may include up to 2 million tonnes placed at the southern end of the Jacobs Ridge portion of the Phase 1 pad to minimize recovery time from the final ore placed on the pad.

18.3 Slaughterhouse Gulch Tailing Storage Facility

The primary flotation tailing disposal facility ("TSF") for the DeLamar project will be located in Sections 30 and 31, Township 4 South, Range 4 West, and Sections 25 and 36, Township 4 South, Range 5 West, in Slaughterhouse Gulch, approximately 6.0 kilometers (3.7 miles) west of the new mill site. Slaughterhouse Gulch is a natural drainage that descends to the south primarily on State and BLM lands. The TSF will be a zoned earth and rockfill embankment that will be located where the valley narrows approximately 1 kilometer (0.6 miles) north of its confluence with Jordan Creek. The Slaughterhouse Gulch TSF will impound flotation tailing that has not been processed by cyanidation and therefore will not be lined in accordance with IDEQ 58.01.013. The earth dam will be designed in accordance with Idaho dam safety regulation IDAPA 37 – DEPARTMENT OF WATER RESOURCES Water Allocations Bureau 37.03.05 -Mine Tailings Impoundment Structures.

18.4 Concentrate Leach Tailing Storage Facility

The concentrate leach tailing storage facility ("CLTSF") will be a smaller, 26-hectare (64.2 acre) impoundment for containment of flotation concentrates from the milling process after they have been leached with cyanide to remove precious metals. To aid in settling, this fine material (P80 of 20 microns) will be blended with a small stream of coarser flotation tailing in roughly a 1:1 blend. The location of this CLTSF is immediately south of the HLP at the head of the unnamed drainage. The construction of the CLTSF in this location will involve placing fill from the Jacobs Ridge pad area to provide initial stormwater storage and then installing a liner system in year 2 that will meet the lining requirements of the IDEQ Rules 58.01.13 – Rules for Ore Processing by Cyanidation.

18.5 Power Generation and Distribution

The electrical power demand at the DeLamar project facilities is currently estimated at 13.5 MW for initial heap-leach process operations, with an additional load of 9.8 MW for the mill circuit. The demand will vary according to the quantity of each ore type to be processed. The average load for the mine is forecast to be 11.6 MW with a peak demand of 23.4 MW. Lifetime electricity consumption is estimated to be 1.8 million MWh.

Existing electrical infrastructure on the project site consists of a 69 kV transmission line operated by Idaho Power Company. Significant upgrades to existing electrical infrastructure would be required to meet the anticipated load increase associated with the project, including construction of new 138 kV transmission lines, substations and tap station upgrades. To reduce capital expenditures of energy infrastructure, ensure power supply resilience and reduce emissions, Integra plans to power the project through an on-site microgrid with a solar electrical generation system and an LNG plant.

This microgrid is planned to consist of a 12 MW solar array that will be installed on the historical tailing impoundment in conjunction with 4.5 MWh of batteries. An LNG power plant is planned to be constructed on site and leased from a third-party provider through a long-term use-based equipment lease.

18.6 Power Pricing

In 2020, the average levelized cost per MWh for contract solar projects nationwide was $24/MWh ($0.024 per kWh) without batteries, and around $40/MWh ($0.04 per kWh) for 90% battery to PV capacity, according to market studies. Based on research conducted by the study team via interviews with utility-scale microgrid developers in the region, a microgrid cost for this site could range from $0.05 to $0.07 per kWh, depending on the renewable fraction. The cost per kWh used for the cash flow model is $0.065 per kWh.

A trade off study was completed to assess capital and operational costs for the microgrid scenario relative to costs associated with upgrading the Idaho Power Company transmission infrastructure to site. The analysis considered both owner-operated on-site electrical generation and a third-party long-term purchasing agreement. The microgrid levelized cost of energy ("LCOE") is estimated at $0.055/kWh, 63% lower than the local electric utility LCOE of $0.149/kWh, which included both the power rate and the capital expense associated with the power line upgrade. With a carbon price of $50/tonne of CO2 added in, the LCOE for the microgrid is $0.065/kWh and $0.168/kWh for grid supplied power.

18.7 Railveyor Haulage System

The project will utilize a Railveyor light rail haulage system to transport ore from the open pits to the crusher facility. The Railveyor system is an autonomous materials haulage system consisting of transport trains, light-rails, electrical drive stations, and materials loading and discharge stations. The system functions similar to a conveyor, but is designed to be modular and relocatable, allowing improved operational flexibility and lower cost. By leveraging the Railveyor system, the DeLamar project has a unique opportunity to realize cost savings compared to typical truck haulage, while reducing its overall fuel consumption and carbon footprint and automating many essential functions that typically would require on-site personnel.

The Railveyor trains will be powered by electric drive stations placed along the track throughout the length of the system. Drive stations consist of 100 hp AC motors geared to commercial truck tires which squeeze the rail cars to propel and stop the train. Because the system will use electric drives to propel rail cars, it will be powered entirely by the onsite electrical system. Railveyor also allows for electricity generation through regenerative braking which results in a net surplus of power generated in the downhill-haul Florida Mountain loop.

18.8 Project Buildings

The proposed heap-leach facility will be located between the DeLamar and Florida Mountain pits. The primary crusher and process facilities will be located just south of the HLPs. Ore will be conveyed from the primary crusher to oxide or non-oxide coarse ore stockpiles accordingly.

Oxide and mixed ore will be conveyed from the oxide coarse ore stockpile to the secondary and tertiary crushing and screening. Crushed oxide and mixed ore may report to the agglomeration circuit or bypass agglomeration if not required. From the agglomeration circuit, the ore will be conveyed to the leach pad via an overland conveyor to a series of grasshopper conveyors that will distribute the ore onto the pad in the prescribed courses. The grasshopper conveyors are not demonstrated on the drawings at this time for clarity as they will be moved throughout the loading of the heap-leach pad. The PLS will flow by gravity to the PLS ponds directly north of the heap-leach pad. An event pond will be located adjacent to the Valley PLS pond to allow for passive overflow if an excessive runoff event occurs. Road access is provided along the east edge of the heap-leach facility to allow access to the ponds. This access route will also serve as a pipe route for PLS and reclaim water piping to be pumped back up to the Merrill-Crowe and process facility pad.

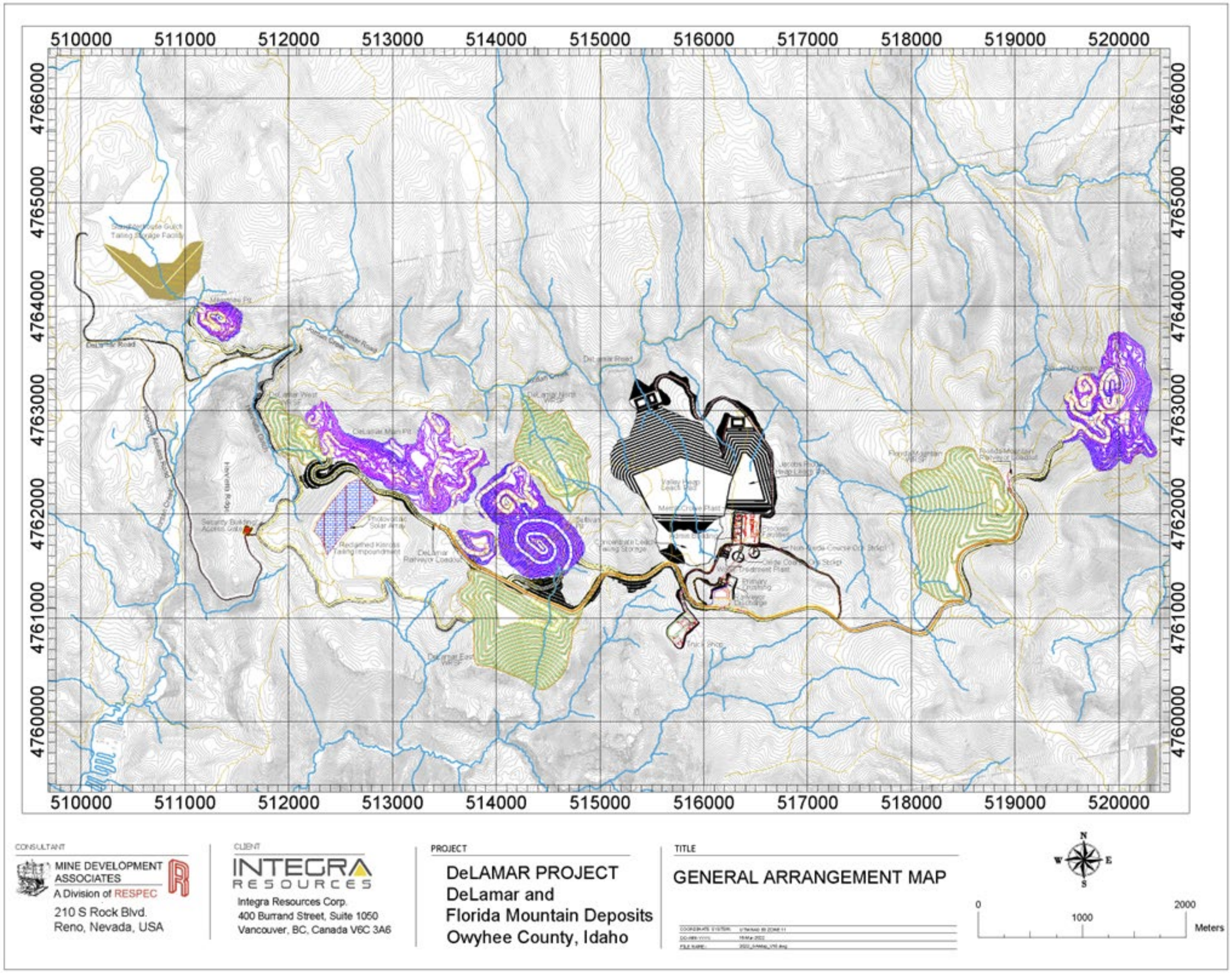
Non-oxide ore will be conveyed from the non-oxide coarse ore stockpile to the mill facility on the process facilities pad. Ore will be subjected to size reduction through the SAG mill and the ball mill. Primary ground ore will pass through one stage of rougher flotation. Concentrate from the flotation cells will be reground, thickened, and sent through a series of agitated leach tanks. PLS from the tank leach circuit will be recovered via CCD cyclone systems, with barren solution from the Merrill-Crowe circuit utilized as the wash solution. The PLS will be clarified in a clarifying thickener and sent to the Merrill-Crowe facility for metal recovery. Post CCD slurry will be thickened, blended to a 1:1 solid mass ratio with flotation tailing slurry, and deposited in the dedicated TSF near the Valley fill leach pad as summarized in Section 18.4. Slurry from the rougher flotation tailing (non-cyanide bearing) will be pumped from the flotation tailing thickener to the west along haul and mine roads to the Slaughterhouse Gulch TSF, approximately 13.3 kilometers (8.4 miles) west of the process facilities pad.

Other buildings located on or near the process facilities pad include the administration/change building, a substation, assay lab, Merrill-Crowe plant, and water treatment plant.

A truck shop is also planned south of the primary crusher and ROM pad. Light vehicle and diesel fuel islands will be constructed just east of the truck shop. A truck wash and tire change pad are also included southeast of the truck shop. Safety and training areas will be provided within the truck shop building. In addition, mine services offices are integral to the truck shop and a laydown yard is proposed directly southwest of the facility. The DeLamar, Milestone, Sullivan, and Florida Mountain pits will be connected to their respective waste dumps and the primary crusher by haul roads.

Figure 18.1 shows the DeLamar and Florida Mountain general arrangement drawing. This includes pit designs, waste-rock storage facilities, heap-leach pads, mill facilities and tailing storage facilities.

Figure 18-1 PFS General Arrangement Drawing



19.0 MARKET STUDIES AND CONTRACTS

No market studies have been undertaken for the PFS. 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 for gold and silver in the months prior to the January 24, 2022 effective date of the reserves and has reviewed current metal prices used in recent NI 43-101 technical reports. In addition, three-year and two-year rolling average gold and silver prices were reviewed along with one-year forward pricing.

As of the end of January 2022, the three-year rolling average gold price based on Kitco metal pricing was $1,669 per ounce. The three-year rolling average silver price was $20.83 per ounce. This compares to the two-year rolling averages of $1,795 and $23.05 per ounce for gold and silver, respectively, at the end of January 2022.

A review of 10 different technical reports from mid- to late-2021 showed gold pricing ranging from $1,500 to $1,750 per ounce gold. Thus, there does not seem to be a lot of consistency on gold prices used. Of these reports, only four projects included silver with ranges of $20.00 to $22.00. Mr. Dyer has reviewed these studies and the rolling average and future prices and has settled on using a mix of consensus one-year future prices along with three-year rolling average prices. This yields a gold price of $1,700 per ounce and a silver price of $21.50 which have been used in the economic analysis discussed in Section 1.0.

Other than royalty contracts as discussed in Section 4, the QP is not aware of any other contracts in place at this time material to the user.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

As discussed in Section 6.0 of this report, the DeLamar area has been mined extensively over the past century and most recently in the 1990s into the early 2000s. Historical mining activities have altered the topography, hydrology, and ecology of the area. Reclamation efforts undertaken at the site by Kinross have been conducted pursuant to federal and state permit requirements and have included waste-rock disposal area reclamation, tailing reclamation efforts, facility removal and cleanup, and surface disturbance reclamation. Integra has primarily conducted site permit compliance, water management activities and exploration since acquisition of the property in 2017.

20.1 Environmental Studies and Permitting

The DeLamar project is located on public lands administered by the BLM, State of Idaho lands, and private lands controlled by Integra in Sections 25, 35, and 36, Township 4 South, Range 5 West (T4S, R5W), Sections 30 and 31, T4S, R3W, Sections 1 through 16, T5S, R3W, Sections 28 through 36 T4S, R4W, Sections 1 through 3, 10, 11, 14, 15, and 22, T5S, R5W, and Sections 1 through 16, T5S, R4W, Boise Base and Meridian. There are several private land parcels within the DeLamar project that are not controlled by Integra with proposed new project activities planned on those parcels. The proposed main access to the DeLamar project is via the Jordan Creek Road from Jordan Valley, Oregon. In general, the proposed mine operations consist of three open pits, four DRSFs, ancillary maintenance and administration facilities, heap leaching facility (including ore-processing operations, a TSF, and a mill processing facility. Integra plans the construction, operation, reclamation, and closing of this mining operation. Major components include:

- Three open pits;
- Four Development Rock Storage Facilities;
- Crushing and conveying system;
- A heap-leach processing facility;
- One tailing storage facilities
- One mill processing facility;
- Reagent area;
- Laydown areas;
- A water treatment plant and distribution system;
- A water extraction and distribution system;
- Potential onsite solar power and LNG power generation system;
- Excess water management system;
- Storm water surface diversion ditches and process water retention basins, and storm water sediment basins;

- Haul roads, ancillary site feature access roads, and ore haulage systems;
- Upgrade of the existing project access road; and
- Truck shop, warehouse, Merrill-Crowe circuit and refinery, fuel storage, guard shack, and laboratory.

Integra proposes to mine approximately 35,000 tonnes per day of heap-leach and 6,000 tonnes per day of mill-grade mineralized material (heap-leach only in Stage 1 and heap-leach and mill in Stage 2). Stage 2 construction would commence in one year at Integra's discretion. The LOM strip ratio would be 2.21 tonnes of waste per one tonne of ore. The life of the operation would be approximately 16 years.

The mineralized material and waste would be extracted from the open pits using conventional surface mining methods of drilling, blasting, loading, and hauling. Integra would use hydraulic shovels or front-end loaders to load the blasted mineralized material and waste into the haul trucks. The haul trucks would transport the waste rock to the rock disposal area near the open pits. The haul trucks would also transport the mineralized material to a load out location, where a Railveyor system would transport the material to the crushing system where the mineralized material would be crushed and delivered to the heap-leach pad, or the mill, for processing using a NaCN solution to leach the precious metals. After milling processing, the residual ore would be placed in a TSF. A Merrill-Crowe process would be used to precipitate the precious metals. The precipitate would then be refined in a furnace to produce doré bars for shipment off site. The project facilities are anticipated to disturb approximately 809 hectares (2,000 acres).

The review and approval process for the Plan of Operations by the BLM constitutes a federal action under the National Environmental Policy Act ("NEPA") and BLM regulations. Thus, for the BLM to process the Plan of Operations, the BLM is required to comply with the NEPA and prepare either an Environmental Assessment ("EA"), or an Environmental Impact Statement ("EIS"). Based on discussions with the BLM, Integra anticipates an EIS will be required to comply with NEPA.

The following sections provide information on historical and recent site characterization efforts, existing environmental conditions, status of project approval and permitting efforts, social and community considerations, proposed mitigation of stream and wetland disturbance, and reclamation and closure activities.

20.1.1 Environmental Baseline Studies

Integra has contracted qualified third parties to perform environmental adequacy reviews of all available existing environmental baseline reports and data compiled from 1979 through present. Additionally, an EA was approved in 1987 for the DeLamar Silver Mine and an EIS was approved in 1995 for the Stone Cabin Mine by previous operators for the site.

In 2020, Integra conducted a technical adequacy audit of all existing environmental information, and began the collection of data that included the following:

- Surface water hydrology and quality;
- Ground water hydrology and quality;

- Geochemistry;
- Water rights; and
- Geotechnical/engineering.

Baseline studies for surface water and groundwater were initiated in spring of 2020. Geotechnical investigations for site features commenced in 2021 and geochemical fieldwork and kinetic testing commenced in 2020 and will continue into Q1 2024. In 2021 to 2023, Integra developed plans of study that included the following:

- Wetland Resources Baseline Plan of Study;
- Aquatic Resources Baseline Plan of Study;
- Wildlife Resources Baseline Plan of Study;
- Class III Cultural Resources Baseline Plan of Study;
- Groundwater Baseline Plan of Study;
- Surface Water Baseline Plan of Study;
- Vegetation Baseline Plan of Study;
- Soils Baseline Plan of Study;
- Noise Baseline Plan of Study;
- Visual Resources Baseline Plan of Study;
- PM10 Quality Assurance Monitoring Plan
- PM2.5 Quality Assurance Monitoring Plan; and
- Geochemical Characterization Plan of Study.

In 2021, Integra, working closely with the BLM and state agencies, completed the review and approval of the initial environmental baseline work plans. In conjunction with the Mine Plan of Operations proposed action, baseline studies were updated in 2022 and 2023 to account for revised and new proposed site features. Baseline surveys initiated in accordance with the 2021 plans of study and continued in 2023 with the updated 2023 plans of study where all baseline studies were completed by the end of the 2023 field season in preparation for filing of the Mine Plan of Operations. All baseline technical reports are underway and anticipated to be completed by the end of 2023/early 2024. Additional studies were undertaken in 2022 and 2023, these include the following:

- Transportation and public safety;
- Recreation and visual;
- Air quality/noise;
- Hazardous materials;
- Reclamation; and
- Socioeconomics.

The data collection and technical reports are scheduled to be completed in the second half of 2023/early 2024. 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.

20.1.2 Federal Permitting

20.1.2.1 Bureau of Land Management Plan of Operations

The Mine Plan of Operations ("MPO") is submitted to the BLM for any surface disturbance in excess of five acres (2.02 hectares). The MPO describes the operational procedures for the construction, operation, and closure of the project. As required by the BLM, the MPO includes a waste-rock management plan, quality assurance plan, a storm water plan, a spill prevention plan, reclamation plan, a monitoring plan, and an interim management plan. In addition, a reclamation report with a Reclamation Cost Estimate ("RCE") for the closure of the project is required. The content of the MPO is based on the mine plan design and the data gathered as part of the environmental baseline studies. The MPO includes all mine and processing design information and mining methods. The BLM determines the completeness of the MPO and, when the completeness letter is submitted to the proponent, the NEPA process begins. The RCE is reviewed by BLM and the bond is determined prior to the BLM issuing a decision on the MPO.

The MPO is planned to be submitted for the project when operational and baseline surveys are complete and operations and design for the project are at a level where an MPO can be developed to the necessary level of detail. Submittal of the MPO is scheduled to be delivered to the BLM in Q4 2023.

20.1.2.2 U.S. Army Corps of Engineers Authorization, Section 404 of Clean Water Act

A permit under Section 404 of the Clean Water Act ("CWA") is required for the discharge of dredged or fill material into waters of the U.S. ("WOTUS"). Dredged or fill material includes tailing, heap-leach facilities, and waste rock. Other activities, in addition to the tailing and waste-rock storage that may require a Section 404 Permit are:

- Road construction;
- Bridges;
- Construction of dams for water storage;
- Stream diversions; and
- Certain reclamation activities.

WOTUS include wetlands. A 2010 U.S. Supreme Court decision found mine tailing to be "fill," and it can therefore be placed in WOTUS with an approval under Section 404 of the CWA.

20.1.2.3 Environmental Impact Statement

Approval of any MPO and Reclamation Plan by the federal agencies for the project as well as accordance with Section 404 requires an environmental analysis under the NEPA. NEPA requires federal agencies to 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 and ore processing, which would disturb over 809 hectares (2,000 acres) of unpatented and patented mining claims, federal (BLM) lands and state lands within the project area and complete reclamation and closure activities, as well as long-term water treatment, to produce silver and gold from mineralized material of the estimated mineral resources. As a result, Integra anticipates that an EIS will be required to meet agency NEPA requirements.

The BLM will be the lead federal agency for the preparation of the EIS, and other agencies will be cooperating agencies. The EIS and associated Record of Decision ("ROD") effectively drives the entire permitting process timeline.

20.1.3 Idaho Permitting

20.1.3.1 Idaho Pollutant Discharge Elimination System Permit ("IPDES")

An IPDES Permit would be required for point source discharges from the mining operation to WOTUS. Likely point discharges would include treated mine drainage, treated net precipitation from the TSFs, 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/or be collected and used as process water or be discharged 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.

20.1.3.2 Other 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 as follows:

- 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 is proposing to process the gold and silver mineralized material at a heap-leach facility and at a mill with associated TSFs. 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. Integra may use underground injection or surface water discharge (as discussed above) rather than land application. Idaho Department of Water Resources ("IDWR") implements the Underground Injection Control program;
- Ground Water Rule – Point of Compliance – 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 need to take into account the high naturally occurring background levels for several parameters;
- Water Rights – There are five decreed water rights associated with the mining activity area. In addition to these established rights, Integra holds three associated permits 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 PFS contemplates relocating one or more unnamed creeks, both temporarily and/or permanently, as part of the overall mine plan;
- Dam Safety Permit – The IDWR requires a Dam Safety Permit for dams greater than 3.05 m (10 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 tailing 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 PFS contemplates construction of a TSF in an unnamed creek within the Slaughterhouse Gulch 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;
- Fuel Storage Facilities – Any proposed fuel storage must also comply with IDEQ design and operating standards, as well as EPA Standards under 40 CFR 112 (may require a Spill Prevention and Countermeasure Plan depending on the size of the facility), 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 and state lands. The Reclamation Plan includes detailed operating plans showing pits, mineral stockpiles, overburdened piles, tailing 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; and
- 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.

20.1.4 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 Southwest 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.

To date, Integra has not entered into any agreements with local communities; however, there have been discussion with the local communities, principally Silver City and Jordan Valley.

20.1.5 Idaho Joint Review Process

The IDL is responsible for implementation of the Idaho Joint Review Process, which was established 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, Section 404 permit, Section 401 Certification of these key permits, and the state Cyanidation permit. 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 EIS/Permitting Timelines and Costs

20.2.1 Permitting Timelines

The discussion below assumes that the BLM is the lead federal agency for NEPA, and that the United States Army Corps of Engineers ("USACE") is 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; USACE, IDEQ, and IDL are cooperating agencies;
- Mining– Estimated at 41,000 tonnes per day (heap-leach and mill) mineralized material with a 2.21:1 waste to mineralized material strip ratio;
- Processing – Tailing by-product (flotation tailing and concentrate leach tailing); heap leaching of ore is also part of the project. Integra has the option to delay mill construction;
- Power – It is anticipated that the existing line power would be utilized along with the development of an on-site solar and LNG electricity generation system to meet further power needs;
- Waste Rock – Some selective placement would be required likely due to potential geochemical reactivity; large volumes would be stored and managed;
- Water Supply – Available from existing or future water rights;
- Water Treatment – Required for surface water discharge, injection, or land application;
- Project Access – Existing Trout Creek Road from Jordan Valley, Oregon with a revised access to the Trout Creek Road within the project area;
- Manpower – Up to 440 direct and indirect jobs during construction; estimated 340 during operations;
- Operating Schedule – Mining and processing year-round; and
- Total Land Disturbance – Over 809 hectares (2,000 acres) of disturbance on public lands, state lands and private lands.

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 – Selection of third-party EIS Contractor, Start baseline confirmatory studies for surface and ground water, geochemistry, aquatic resources, wildlife, cultural resources, vegetation, and soils as well as air quality and wetlands work. This work commenced in 2021. Negotiate an MOU with the BLM for preparing the EIS. Conduct initial internal scoping with "high up" agency and political contacts;
- Months 24 to 48 – Commence preparation of the Initial Plan of Operations. Concurrently, develop all other permit applications for submittal. The third-party contractor would 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 as required under NEPA, but also meet the USACE's alternative analysis requirements. Integra would file its Initial Plan of Operations with the BLM later in this window at around Months 36 to 48, which would trigger the EIS. This is currently anticipated for Q4 2023 and Q1 2024;
- The BLM would publish the Notice of Intent to prepare an EIS during this timeframe;
- Integra plans to complete a feasibility study during the timeline of the proposed 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 or alternation needed for the project;

- Months 42 to 60 – 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, cannot be issued until after the final EIS and ROD by the BLM have been issued;
- Months 60 to 64 – A draft EIS would be produced for public review. The review period would be about 60 days;
- At Months 64 to 76, the final EIS would be completed by the BLM. 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 78. The remaining permits would also be issued over this period; and
- Summary Best Case Schedule – Estimated at 76 to 78 months (6.5 years) from project inception to issuance of final ROD with a concurrent baseline data collection program. Integra is currently at month 40 of this preliminary proposed timeline.

20.2.2 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" – $1.9M (represents all costs for third-party EIS and BLM reimbursement);
- Support Engineering – $1.0M (does not include feasibility study);
- Legal – $0.5M;

- Potential Baseline Study Needs – $7.5M to $9.5M, depending on ground water and geochemistry issues;
- Permitting "Project" – $3.5M (represents all costs for separate permitting program); and
- Total Estimate – $14.4M to $16.4M.

20.2.3 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.

20.3 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 Dark Skies Compliant Lighting

In order to limit light pollution, the Dark Sky Reserve has created a detailed Lightscape Management Plan. Important considerations include meeting lumen and temperature requirements, shielding fixtures, and reducing glare. The DeLamar site does not fall within a designated Dark Sky area, but the following will be considered in future engineering studies:

1. Dark Sky compliance states that any light emitting over 500 initial lumens (40 candela) must be shielded to prevent any light from emitting beyond horizontal. Dark Sky regulations require all lights to be a maximum of 3000k; and
    a. Lighting fixtures should be installed high and face vertically down. Directional lighting is safer for workers and would drastically reduce the amount of light pollution emitted from a site.

20.5 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 metal leaching ("ML") or acid rock drainage ("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 tailing material from mineral benefaction processes, and determination of final pit wall geochemistry.

Geochemical characterization is an iterative process and geochemical samples were collected from historic drill core that was available onsite as well as concurrent exploration, metallurgical, and geotechnical drilling that was conducted from 2020 to 2022. Static and kinetic testing was conducted in 2021 and 2022 and the results of the program were used to inform reclamation and closure activities as well as waste rock placement for WRSF's and was used as the framework modeling for fate and transport modeling.

20.6 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 monitoring the effect of temperature on fish species that exist in the Jordan Creek drainage, as well as related environmental closure needs required to maintain 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. Integra would seek to develop an economic mine plan and closure/reclamation strategy that integrates habitats and restoration components. The plan would meet all standards of the Clean Water Act. The plan would also mitigate wetland impacts to recreate, enhance, or replace productive wetlands and other riparian habitats. It is anticipated that the reclamation and closure of the tailing and heap-leach facilities would consist of fluid management through evaporation or treatment and discharge, covering the facilities growth media, and then revegetating. The estimated reclamation costs for the project, using the Nevada Standardized Reclamation Cost Estimator is approximately $30.8 million. If Integra chooses to delay the construction and operation of the mill facility, the estimated reclamation cost could be approximately $24.8 million.

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;
- Restricting or eliminating the migration of potential contaminants of concern from all sources based on the proposed mine plan;
- Restricting or eliminating potential public safety risks associated with the potential decommissioned and reclaimed mine site;

- Restoring the property, to the extent possible, to the current pre-mining conditions; and
- Improving the property by incorporating environmental mitigation projects as identified through the NEPA process.

21.0 CAPITAL AND OPERATING COSTS

Capital and operating costs were estimated for the PFS by RESPEC (mining and infrastructure), M3 Engineering (both heap leaching and milling processing), and Mr. John Welsh, senior principal of Welsh Hagan Associates (leach pads and tailing impoundments). Table 21.1 summarizes the estimated capital costs for the project. The LOM total capital costs are estimated as $589.5 million, including $307.6 million in preproduction and $281.8 million for expansion and sustaining capital. Sustaining capital includes $30.8 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	Sustaining Yr 1 to Yr 17	Total LOM
Mining Equipment	$ 28,859	$ 88,544	$ 117,403
Pre-Stripping	$ 12,712	$ -	$ 12,712
Other Mine Capital	$ 1,919	$ 225	$ 2,144
Sub-Total Mine	$ 43,490	$ 88,769	$ 132,260
Processing			
Leach Pad Construction Cost	$ 42,296	$ 11,035	$ 53,331
Oxide Plant Construction	$ 165,198	$ 8,842	$ 174,040
Non Oxide Mill Construction	$ -	$ 132,005	$ 132,005
Tailings Storage Facility Construction	$ 3,836	$ 58,793	$ 62,629
Sub-Total Processing	$ 211,330	$ 210,675	$ 422,005
Infrastructure			
Power	$ 3,500	$ -	$ 3,500
Access Road	$ 8,957	$ -	$ 8,957
Other	$ 7,652	$ 974	$ 8,626
Sub-Total Infrastructure	$ 20,109	$ 974	$ 21,083
Owner's Costs	$ 7,001	$ -	$ 7,001
SUB-TOTAL	$ 281,930	$ 300,418	$ 582,349
Other			
Working Capital	$ 19,518	$ (19,518)	$ -
Cash Deposit for Reclamation Bonding	$ 6,167	$ (6,167)	$ -
Salvage Value	$ -	$ (23,729)	$ (23,729)
TOTAL	$ 307,615	$ 251,004	$ 558,620
Reclamation	$ -	$ 30,835	$ 30,835
Total Including Reclamation Costs	$ 307,615	$ 281,839	$ 589,454

Notes:
1. *Capital costs include contingency and EPCM costs;*
2. *Mining equipment includes the cost of Railveyor.*
3. *Major mining equipment assumes financing by equipment vendor with 10% down and principal payments included under sustaining capital column and interest payments included in operating costs;*
4. *Sustaining capital showed in this table includes expansion capital (non-oxide plant) and principal payment of mining equipment leases (see note 3 above);*
5. *Working capital is returned in year 17;*
6. *Cash deposit = 20% of bonding requirement. Released once reclamation is completed; and*
7. *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 $12.93 per tonne processed for the LOM. This includes mining costs which are estimated to be $1.90 per tonne mined. The total cash cost is estimated to be $923 per ounce of gold equivalent and site-level, all-in sustaining costs are estimated to be $955 per ounce of gold equivalent.

Table 21-2 Operating and Total Cost Summary

	US/Tonne	
LOM Operating Costs	**Mined**	**Processed**
Mining	$ 1.90	$ 6.09
Processing (HL + Mill)		$ 5.99
G&A		$ 0.86
Total Site Costs		**$ 12.93**
LOM Cash Costs and Site Level All-in Sustaining Costs	**By-Product** (1)	**Co-Product** (2)
Mining	$ 647	$ 418
Processing	$ 640	$ 414
G&A	$ 92	$ 59
Total Site Costs	**$ 1,379**	**$ 891**
Transport & Refining	$ 27	$ 17
Royalties	$ 23	$ 15
Total Cash Costs	**$ 1,429**	**$ 923**
Silver By-Product Credits	$ (931)	$ -
Total Cash Costs Net of Silver by-Product	**$ 498**	**$ 923**
Sustaining Capital	$ 50	$ 32
Site Level All-in Sustaining Costs	**$ 548**	**$ 955**

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 were based on estimation guides, quotations from equipment vendors 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	Pre-Prod	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Total
Primary Mining Equipment	K USD	$ 3,191	$ 7,917	$ 7,877	$ 7,852	$ 7,942	$ 8,346	$ 8,771	$ 9,218	$ 9,688	$ -	$ 70,802
Rail-Veyor	K USD	$ 24,875	$ -	$ -	$ -	$ 1,886	$ 9,429	$ -	$ -	$ -	$ -	$ 36,190
Support Equipment	K USD	$ 665	$ 1,058	$ 877	$ 921	$ 968	$ 1,018	$ 1,070	$ 1,124	$ 1,181	$ -	$ 8,882
Blasting Equipment	K USD	$ 33	$ 31	$ 32	$ 34	$ 35	$ 37	$ 39	$ 41	$ 43	$ -	$ 325
Mine Maintenance Eqipment	K USD	$ 97	$ 137	$ 119	$ 125	$ 131	$ 138	$ 145	$ 152	$ 160	$ -	$ 1,205
Other Mine Capital	K USD	$ 1,919	$ 17	$ 6	$ 2	$ -	$ -	$ 200	$ -	$ -	$ -	$ 2,144
Total Mine Equipment Capital	K USD	$ 30,778	$ 9,159	$ 8,911	$ 8,934	$ 10,963	$ 18,969	$ 10,225	$10,536	$11,073	$ -	$119,548
Prestripping Costs	K USD	$ 12,712	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 12,712
Total Mine Capital	K USD	$ 43,490	$ 9,159	$ 8,911	$ 8,934	$ 10,963	$ 18,969	$ 10,225	$10,536	$11,073	$ -	$132,260

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 $70.8 million which includes:

- $620,000 for a pioneering drill;
- $8.5 million for production drills;
- $2.5 million for a large loader;
- $11.9 million for hydraulic shovels; and
- $47.3 million for 136-tonne capacity haul trucks.

Primary equipment costs assume financing with the following terms:

- Cash deposit upon purchase: 10%
- Annual interest rate included in operating cost: 5%
- Financing terms: 5-years

21.1.2 Railveyor

The cost estimate to install a Railveyor system to facilitate ore haulage from the edge of the pit to the process facility is estimated at $36.2 million including the following:

- $9.8 million for train cars;
- $20.0 million for drive stations;
- $3.3 million for track;
- $2.0 million for feeders;
- $535,000 for installation and commissioning; and
- $430,000 for control systems.

21.1.3 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 $8.9 million and includes:

- $4.4 million for dozers of various sizes;
- $2.5 million for motor graders;
- $1.4 million for water trucks;
- $33,000 for in-pit pumps to control runoff water;

- $557,000 for a 50-ton capacity crane (to be shared between mining and process); and
- $67,000 for a flatbed truck used for moving maintenance items within the mine.

21.1.4 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 $325,000 which includes $256,000 for one explosives truck and $70,000 for a skid loader.

21.1.5 Mine Maintenance Capital

The cost estimate for mine maintenance capital is $1.2 million, which includes one large lubrication/fuel truck at $541,000, two mechanic's trucks totaling $472,000, and one tire truck totaling $192,000.

21.1.6 Other Capital

Other capital includes an assortment of equipment and facilities totaling $2.1 million. This includes:

- $110,000 for light plants;
- $200,000 preparation for explosives storage site;
- $86,000 for ANFO storage bins;
- $15,000 for powder magazines to store boosters;
- $8,000 for a cap magazine;
- $64,000 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;
- $86,000 for fuel island facilities;
- $400,000 in access roads to each deposit and site preparation; and
- $150,000 for ambulance and firefighting equipment.

The access roads and shop are described in Section 18.1 and Section 18.8, respectively.

21.1.7 Mine Preproduction Costs

Mine preproduction costs are considered as the cost of all mining prior to the start of metal production from the leach pad. For the PFS this is a 12-month period during which there are six months of mining. The mining costs during preproduction total $12.7 million.

21.2 Process Capital

The process plant capital costs were developed for the initial phase, oxide, and mixed ore processing circuit, as well as for the Stage 2 non-oxide ore processing circuit. The capital costs for each phase are comprised of direct costs and indirect costs. The direct costs were developed from labor, materials, plant equipment, sub-contracts, and construction equipment. Freight is also included with the direct costs. Indirect costs were applied to the direct costs to account for items such as: construction support, EPCM, vendor support during specialty construction and commissioning, spare parts, contingency, owner's costs, and taxes. Capital costs were estimated based on 4th quarter 2021 US dollars and are presented with no added escalation.

M3 Engineering developed the costs for site layout within the process area, the process plant, and several ancillaries. The Stage 1 (Phase 1) process plant (oxide ore) includes the three-stage crushing circuit, ore conveyance and stacking on the heap-leach pads, heap-leach pad solution systems, the Merrill-Crowe gold/silver recovery plant and refinery, and ancillaries (administration building, warehouse, truck shop, maintenance, and metallurgy and assay lab). Stage 1 (Phase 2) process plant (oxide ore) includes agglomeration and additional conveyors for ore stacking on the pad. The Stage 2 process plant (non-oxide ore) includes primary crushed non-oxide ore conveyance, grinding, flotation, concentrate regrind, concentrate leach, concentrate leach counter-current decantation, tailing slurry dewatering, and tailing slurry transport to independent tailing impoundments (rougher flotation tailing and concentrate leach tailing). Welsh-Hagen developed the costs for the heap-leach pads, as well as the tailing impoundments and included freight, sales taxes, EPCM, and contingencies to their estimates.

Indirect costs were then calculated following industry accepted methodologies, including application of contingency based on the completed level of design on a scope or individual work type basis. The agglomerate contingency for the process plant is estimated at 20% of total contracted cost for each phase. Total contracted costs include all process plant direct costs, plus construction support costs, EPCM costs, vendor support costs, spare parts costs, and county taxes. First fills were calculated by M3 Engineering. Owner's Costs were defined by Integra and are carried outside of the process plant estimates. Owyhee County sales taxes were included at 6% of plant equipment and material costs. Process plant capital costs estimates were based on the purchase of new equipment.

The total evaluated process plant capital costs are projected to be in the accuracy range of -20% / +25%. Stage 1 (Phase 1) process plant capital costs for the oxide and mixed ore types are summarized in Table 21.4. Stage 1 (Phase 2) process plant capital costs for the oxide and mixed ores are summarized in Table 21.5. Stage 2 process plant capital costs for the non-oxide ore are summarized in Table 21.6.

Table 21-4 Stage 1 (Phase 1) Capital Costs, Oxide & Mixed Ore Heap Leach/Merrill Crowe Plant

Category (all costs are in USD 1,000)	Labor	Plant Equip.	Material	Sub Contract	Const. Equip.	Total
General Site (Earthworks)	755	-	119	-	376	1,251
Primary Crushing	5,122	4,938	3,501	-	1,234	14,796
Fine Crushing	10,863	20,204	6,036	-	2,204	39,307
Conveying/Stacking	1,650	6,525	1,004	-	59	9,239
Heap Leach Solution Systems	2,466	1,825	3,028	-	97	7,417
Merrill Crowe	4,304	7,595	2,692	-	277	14,868
Refinery	628	536	428	-	69	1,661
Water Systems (In-Plant)	875	2,504	467	-	45	3,891
Reagents	156	337	74	-	10	576
Ancillaries	2,522	983	6,323	-	930	10,757
Freight	-	3,636	1,894	-	-	5,530
Sub-Total Direct Cost	**29,343**	**49,082**	**25,566**	**-**	**5,301**	**109,292**
Contractor Mobilization						1,948
EPCM						18,355
Vendor Support						1,472
Spare Parts (Capital, Comm.)						2,518
First Fills						750
Owner's Cost						-
Taxes (County)						4,147
Contingency						26,717
Sub-Total Indirect Cost						**55,907**
TOTAL CAPITAL COST						**165,198**

Table 21-5 Stage 1 (Phase 2) Capital Costs, Oxide & Mixed Ore Heap Leach/Merrill Crowe Plant

Category (all costs are in USD 1,000)	Labor	Plant Equip.	Material	Sub Contract	Const. Equip.	Total
Agglomeration	960	1,755	649	-	62	3,425
Conveying/Stacking	375	1,375	301	-	8	2,059
Freight	-	250	76	-	-	326
Sub-Total Direct Cost	**1,335**	**3,380**	**1,026**	**-**	**70**	**5,810**
Contractor Mobilization						93
EPCM						974
Vendor Support						101
Spare Parts (Capital, Comm.)						186
First Fills						-
Owner's Cost						-
Taxes (County)						245
Contingency						1,433
Sub-Total Indirect Cost						**3,032**
TOTAL CAPITAL COST						**8,842**

Table 21-6 Stage 2 Capital Costs – Non-Oxide Ore Grinding, Flotation, Leach Plant

Category (all costs are in USD 1,000)	Labor	Plant Equip.	Material	Sub Contract	Const. Equip.	Total
General Site	190	-	-	-	-	190
Primary Crushed Ore Conveyance	1,442	2,178	1,052	-	161	4,833
Crushed Ore Reclaim	2,241	1,159	1,144	-	304	4,848
Grinding	9,528	12,545	6,442	-	820	29,335
Flotation & Concentrate Regrind	5,988	7,081	3,705	-	573	17,347
Concentrate Leaching	2,496	2,352	1,749	-	253	6,850
Concentrate CCD	1,341	2,733	772	-	66	4,912
Flotation Tailing Thickening	4,431	1,669	2,567	-	993	9,660
Water Systems (In-Plant)	380	978	214	-	16	1,589
Reagents	1,062	1,965	601	-	74	3,702
Freight	-	2,613	1,460	-	-	4,072
Sub-Total Direct Cost	**29,101**	**35,271**	**19,707**	**-**	**3,260**	**87,339**
Contractor Mobilization						1,433
EPCM						14,647
Vendor Support						1,058
Spare Parts (Capital, Comm.)						1,940
First Fills						1,250
Owner's Cost						-
Taxes (County)						3,054
Contingency						21,283
Sub-Total Indirect Cost						**44,666**
TOTAL CAPITAL COST						**132,005**

21.2.1 Freight

Estimates for equipment and material freight costs are based on bulk freight loads and have been estimated at 8% of the equipment and material costs.

21.2.2 Construction Support

Mobilization is included as an indirect cost at 5% of total direct field costs for civil contracts and 1.5% for all other total direct field costs for process plant direct costs.

Contractor temporary construction facilities are included with Owner's Costs. Temporary construction power is included at 0.1% of total direct field cost.

21.2.3 EPCM

Engineering is included at 6.0% of total constructed cost ("TCC") for the process plant scope. Management and accounting are included at 0.75% of TCC. Project services are included at 1.0% of TCC. Project controls are included at 0.75% of TCC. Construction management cost is included at 6.5% of TCC. An EPCM Fee is included at 1.5% of total direct field cost. EPCM construction trailers are expected to be shared with Owner trailers and are included as part of the Owner's Costs.

21.2.4 Vendor Support

Vendor supervision of specialty construction is included at 1% of plant equipment supply costs. Vendor pre-commissioning and vendor commissioning are each included at 1% of plant equipment supply costs.

21.2.5 Spare Parts

Capital spare parts are included at 5.0% of plant equipment supply costs. Commissioning spare parts are included at 0.5% of plant equipment supply costs. Two-year operating spare parts are excluded.

21.2.6 Heap-Leach Pad Capital

The estimated cost of the Phase 1 portion of the HLP is $42.3 million and the estimated cost of the Phase 2 portion is $11 million. These costs include contractor mobilizations, earthwork, synthetic liner purchase and installation, freight on materials, sales taxes at 6% of direct cost, EPCM at 4% of direct cost, and 20% for contingency.

21.2.7 Tailing Impoundments

The estimated cost of the Phase 1 portion of the Slaughterhouse Gulch TSF is primarily for earthwork to construct the starter embankment and is $24.7 million. Later stages to raise the embankment are estimated to cost $29.8 million in years 5 and 10 of operation. These estimates include contractor costs, sales taxes on materials at 6%, and 20% contingency.

The estimated cost of the CLTSF includes earthwork and purchase and installation of HDPE liner. The estimated cost is $8.1 million. The estimate includes freight and sales taxes at 6% on the synthetic liner, EPCM at 4% of direct cost, and 20% contingency.

21.3 Owner and Infrastructure Capital Costs

RESPEC estimated owner and infrastructure capital costs to be $28.1million for the LOM. This includes costs for power, water, access, security, snow removal equipment, warehouse and offices, light vehicles, and preproduction costs, and excludes preproduction mining costs. The estimated costs are shown in Table 21.7 with further information as follows:

- Power distribution – $3.5 million based on discussions with Agreeko and micro grid provider;
- Site water and distribution –$150,000 for two water wells and some water distribution piping;
- Water treatment plant - $6,100,000 to treat potential contact water discharge;
- Access road – $8,957,000 for road improvements and access road between deposits (including security and fencing);
- Security & Fencing - $100,000;
- Snow removal equipment - $150,000 for a sanding truck with plow;
- Light vehicles - $2.1 million (see Section 21.3.1); and

- Preproduction G&A and process costs - $7.0 million (see Section 21.3.2).

Note: preproduction costs related to mining (pre-stripping) have been included in mining capital (see Section 21.1.7).

Table 21-7 Infrastructure & Owners Capital

Infrastructure	Units	Pre-Prod	Yr_5	Total
Power & Substation	K USD	3,500	-	3,500
Site Water	K USD	150	-	150
Water Treatment Plant	K USD	6,100	-	6,100
Access Road	K USD	8,957	-	8,957
Security & Fencing	K USD	100	-	100
Snow Removal Equipment	K USD	150	-	150
Light Vehicles	K USD	1,152	974	2,126
Total Site Infrastructure	K USD	20,109	974	21,083
Preporduction Costs (Less Mining Preproduction)				
Preproduction G&A	K USD	5,157	-	5,157
Preproduction Process	K USD	1,843	-	1,843
Total Preproduction Costs	K USD	7,001	-	7,001
Total Owners and Other Capital	K USD	27,110	974	28,084

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 21.8. This estimate includes $1,152,000 in initial costs and $974,000 in replacement costs occurring in year 5.

21.3.2 Preproduction Owner Costs

Preproduction owner's costs include G&A and some processing costs during preproduction. Costs related to preproduction mining (pre-stripping) have been included in mining capital (see Section 21.1.7).
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 $5.2 million.

Preproduction process costs are estimated to be $1.8 million. 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.

Table 21-8 Light Vehicle Cost Estimate

	Units	Pre-Prod	Yr_5	Total
Administration				
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

21.4 Reclamation Costs and Salvage Value

Reclamation costs were estimated to be approximately $30.8 million at the end of the mine life. The reclamation costs were estimated using the Standardized Reclamation Cost Estimator (version 1.4). 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 $22.8 million were estimated with an additional $8.0 million in indirect costs.

The reclamation costs were assumed to be secured with a surety bond. A required cash collateral deposit of $6.2 million (20%) was assumed for the surety bond and included in preproduction 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 $6.9 million in year 17 for process equipment related to leaching, and $5.2 million in year 17 for milling equipment.

A total of $23.7 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. In the case of the Railveyor system, a 10% salvage value was applied at the end of use. The mining equipment salvage credit was taken in year 17.

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 and category in Table 21.9. Note that while the costs for preproduction are shown in the cost tables below, these costs are capitalized as preproduction costs. Before capitalization of preproduction costs, the LOM mining costs are $756.1 million, or $1.91/tonne mined. This includes $0.04/tonne for lease interest charges.

Table 21-9 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Mine General Service	K USD	$ 445	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 826	$ 56	$ -	$ 13,973
Mine Maintenance	K USD	$ 1,351	$ 2,791	$ 2,816	$ 2,818	$ 2,816	$ 2,816	$ 2,816	$ 2,818	$ 2,816	$ 2,816	$ 2,816	$ 2,818	$ 2,077	$ 2,077	$ 2,077	$ 2,079	$ 2,066	$ 169	$ -	$ 42,855
Engineering	K USD	$ 424	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 501	$ 501	$ 501	$ 501	$ 480	$ 20	$ -	$ 11,660
Geology	K USD	$ 464	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 524	$ 524	$ 524	$ 524	$ 514	$ 33	$ -	$ 12,871
Drilling	K USD	$ 1,120	$ 4,650	$ 6,412	$ 7,025	$ 7,626	$ 7,064	$ 6,185	$ 6,130	$ 5,996	$ 5,996	$ 5,996	$ 3,366	$ 2,223	$ 1,617	$ 1,220	$ 1,009	$ 888	$ 56	$ -	$ 74,578
Blasting	K USD	$ 1,096	$ 5,208	$ 7,265	$ 7,714	$ 8,665	$ 7,863	$ 6,934	$ 6,923	$ 6,775	$ 6,775	$ 6,775	$ 3,916	$ 2,530	$ 1,905	$ 1,472	$ 1,191	$ 983	$ 50	$ -	$ 84,040
Loading	K USD	$ 1,555	$ 7,695	$ 10,067	$ 10,706	$ 11,178	$ 10,504	$ 9,739	$ 10,235	$ 9,771	$ 9,801	$ 9,815	$ 5,637	$ 3,420	$ 1,086	$ 866	$ 733	$ 653	$ 227	$ -	$ 113,687
Hauling	K USD	$ 4,124	$ 17,670	$ 25,190	$ 33,003	$ 29,442	$ 32,655	$ 26,565	$ 22,306	$ 18,784	$ 20,856	$ 21,306	$ 14,883	$ 10,864	$ 7,863	$ 5,801	$ 4,993	$ 4,169	$ 1,388	$ -	$ 301,862
Mine Support	K USD	$ 2,133	$ 6,106	$ 6,106	$ 6,116	$ 6,106	$ 6,106	$ 6,106	$ 6,116	$ 6,106	$ 6,106	$ 6,106	$ 4,130	$ 3,465	$ 3,465	$ 3,465	$ 3,470	$ 3,465	$ 292	$ -	$ 84,967
Total Mining Before Leasing	K USD	$ 12,712	$ 46,645	$ 60,381	$ 69,907	$ 68,357	$ 69,532	$ 60,869	$ 57,054	$ 52,772	$ 54,874	$ 55,338	$ 37,275	$ 26,447	$ 19,881	$ 16,768	$ 15,344	$ 14,044	$ 2,292	$ -	$ 740,492
Equipment Lease Interest Payments	K USD	$ -	$ 2,928	$ 3,058	$ 2,784	$ 2,344	$ 1,882	$ 1,396	$ 885	$ 348	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 15,625
Total Mining Cost	K USD	$ 12,712	$ 49,572	$ 63,439	$ 72,691	$ 70,702	$ 71,414	$ 62,265	$ 57,939	$ 53,120	$ 54,874	$ 55,338	$ 37,275	$ 26,447	$ 19,881	$ 16,768	$ 15,344	$ 14,044	$ 2,292	$ -	$ 756,117
Cost per Ton																					
Mine General Service	$/t	$ 0.10	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.08	$ 0.11	$ 0.16	$ 0.22	$ 0.30	$ 0.87	$ -	$ 0.04
Mine Maintenance	$/t	$ 0.30	$ 0.11	$ 0.08	$ 0.07	$ 0.07	$ 0.07	$ 0.08	$ 0.08	$ 0.09	$ 0.09	$ 0.09	$ 0.16	$ 0.19	$ 0.28	$ 0.39	$ 0.54	$ 0.75	$ 2.63	$ -	$ 0.11
Engineering	$/t	$ 0.10	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.04	$ 0.05	$ 0.07	$ 0.09	$ 0.13	$ 0.17	$ 0.32	$ -	$ 0.03
Geology	$/t	$ 0.10	$ 0.04	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.05	$ 0.07	$ 0.10	$ 0.14	$ 0.19	$ 0.52	$ -	$ 0.03
Drilling	$/t	$ 0.25	$ 0.19	$ 0.19	$ 0.19	$ 0.18	$ 0.18	$ 0.18	$ 0.18	$ 0.18	$ 0.18	$ 0.18	$ 0.19	$ 0.21	$ 0.21	$ 0.23	$ 0.26	$ 0.32	$ 0.87	$ -	$ 0.19
Blasting	$/t	$ 0.25	$ 0.21	$ 0.21	$ 0.20	$ 0.20	$ 0.20	$ 0.21	$ 0.21	$ 0.21	$ 0.21	$ 0.21	$ 0.22	$ 0.23	$ 0.25	$ 0.28	$ 0.31	$ 0.36	$ 0.78	$ -	$ 0.21
Loading	$/t	$ 0.35	$ 0.31	$ 0.29	$ 0.28	$ 0.26	$ 0.27	$ 0.29	$ 0.30	$ 0.30	$ 0.30	$ 0.30	$ 0.31	$ 0.32	$ 0.14	$ 0.16	$ 0.19	$ 0.24	$ 3.53	$ -	$ 0.29
Hauling	$/t	$ 0.93	$ 0.72	$ 0.73	$ 0.87	$ 0.69	$ 0.85	$ 0.79	$ 0.66	$ 0.57	$ 0.63	$ 0.65	$ 0.83	$ 1.01	$ 1.04	$ 1.10	$ 1.31	$ 1.52	$ 21.58	$ -	$ 0.76
Mine Support	$/t	$ 0.48	$ 0.25	$ 0.18	$ 0.16	$ 0.14	$ 0.16	$ 0.18	$ 0.18	$ 0.19	$ 0.19	$ 0.19	$ 0.23	$ 0.32	$ 0.46	$ 0.66	$ 0.91	$ 1.26	$ 4.54	$ -	$ 0.21
Total Mining Before Leasing	$/t	$ 2.86	$ 1.89	$ 1.75	$ 1.85	$ 1.60	$ 1.81	$ 1.81	$ 1.70	$ 1.61	$ 1.67	$ 1.68	$ 2.07	$ 2.45	$ 2.64	$ 3.17	$ 4.01	$ 5.12	$ 35.64	$ -	$ 1.87
Equipment Lease Payments	$/t	$ -	$ 0.12	$ 0.09	$ 0.07	$ 0.05	$ 0.05	$ 0.04	$ 0.03	$ 0.01	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 0.04
Total Mining Cost	$/t	$ 2.86	$ 2.01	$ 1.83	$ 1.93	$ 1.66	$ 1.85	$ 1.85	$ 1.72	$ 1.62	$ 1.67	$ 1.68	$ 2.07	$ 2.45	$ 2.64	$ 3.17	$ 4.01	$ 5.12	$ 35.64	$ -	$ 1.91

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 21.10 shows the yearly cost estimate for the mine general services.

Table 21-10 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Supervision	K USD	$ 367	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 662	$ 42	$ -	$ 11,214
Hourly Personnel	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total	K USD	$ 367	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 676	$ 662	$ 42	$ -	$ 11,214
Engineering																					
Salaried Personnel	K USD	$ 248	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 442	$ 304	$ 304	$ 304	$ 304	$ 290	$ 12	$ -	$ 6,622
Hourly Personnel	K USD	$ 161	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 322	$ 167	$ 167	$ 167	$ 167	$ 160	$ 6	$ -	$ 4,535
Total	K USD	$ 409	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 763	$ 471	$ 471	$ 471	$ 471	$ 449	$ 18	$ -	$ 11,156
Mine Geology																					
Salaried Personnel	K USD	$ 83	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 114	$ -	$ -	$ 2,060
Hourly Personnel	K USD	$ 363	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 726	$ 363	$ 363	$ 363	$ 363	$ 363	$ 30	$ -	$ 10,198
Total	K USD	$ 446	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 850	$ 487	$ 487	$ 487	$ 487	$ 477	$ 30	$ -	$ 12,258
Supplies & Other																					
Mine General Services Supplies	K USD	$ 6	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 1	$ -	$ 202
Engineering Supplies	K USD	$ 15	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 30	$ 3	$ -	$ 504
Geology Supplies	K USD	$ 19	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 37	$ 3	$ -	$ 614
Software Maintanance & Support	K USD	$ 15	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 29	$ 2	$ -	$ 481
Outside Services	K USD	$ 38	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 6	$ -	$ 1,244
Office Power	K USD	$ 5	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 1	$ -	$ 169
Light Vehicles	K USD	$ 29	$ 69	$ 69	$ 70	$ 69	$ 69	$ 69	$ 70	$ 69	$ 69	$ 69	$ 70	$ 69	$ 69	$ 69	$ 70	$ 67	$ 6	$ -	$ 1,145
Total	K USD	$ 126	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 263	$ 261	$ 22	$ -	$ 4,358
Totals - Mining General																					
Mine General	K USD	$ 445	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 843	$ 826	$ 56	$ -	$ 13,973
Engineering	K USD	$ 424	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 794	$ 501	$ 501	$ 501	$ 501	$ 480	$ 20	$ -	$ 11,660
Geology	K USD	$ 464	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 887	$ 524	$ 524	$ 524	$ 524	$ 514	$ 33	$ -	$ 12,871
Totals	K USD	$ 1,334	$ 2,524	$ 2,524	$ 2,524	$ 2,524	$ 2,524	$ 2,524	$ 2,524	$ 2,524	$ 2,524	$ 2,524	$ 2,524	$ 1,868	$ 1,868	$ 1,868	$ 1,869	$ 1,820	$ 110	$ -	$ 38,505
Cost per Ton Mined																					
Mine General	$/t	$ 0.10	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.08	$ 0.11	$ 0.16	$ 0.22	$ 0.30	$ 0.87	$ -	$ 0.04
Engineering	$/t	$ 0.10	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.04	$ 0.05	$ 0.07	$ 0.09	$ 0.13	$ 0.17	$ 0.32	$ -	$ 0.03
Geology	$/t	$ 0.10	$ 0.04	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.05	$ 0.07	$ 0.10	$ 0.14	$ 0.19	$ 0.52	$ -	$ 0.03
Totals	$/t	$ 0.30	$ 0.10	$ 0.07	$ 0.07	$ 0.06	$ 0.07	$ 0.07	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.14	$ 0.17	$ 0.25	$ 0.35	$ 0.49	$ 0.66	$ 1.71	$ -	$ 0.10

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 21.11. Note that these costs do not include the maintenance labor directly allocated to the various equipment, which is accounted for in other mining cost categories.

Table 21-11 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Supervision	K USD	$ 276	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 511	$ 500	$ 32	$ -	$ 8,467
Planners	K USD	$ 77	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 77	$ 77	$ 77	$ 77	$ 77	$ 6	$ -	$ 2,172
Hourly Personnel	K USD	$ 662	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 1,324	$ 662	$ 662	$ 662	$ 662	$ 662	$ 55	$ -	$ 18,594
Total	K USD	$ 1,015	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,250	$ 1,250	$ 1,250	$ 1,250	$ 1,240	$ 94	$ -	$ 29,233
Other Costs																					
Supplies	K USD	$ 72	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 144	$ 12	$ -	$ 2,388
Light Vehicles	K USD	$ 9	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 21	$ 7	$ -	$ 358
Total	K USD	$ 81	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 165	$ 19	$ -	$ 2,746
Consumables & Other Costs	K USD	$ 285	$ 681	$ 703	$ 705	$ 703	$ 703	$ 703	$ 705	$ 703	$ 703	$ 703	$ 705	$ 703	$ 703	$ 703	$ 705	$ 703	$ 65	$ -	$ 11,582
Parts / MARC Cost	K USD	$ 51	$ 120	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 11	$ -	$ 2,039
Wages & Salaries	K USD	$ 1,015	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,989	$ 1,250	$ 1,250	$ 1,250	$ 1,250	$ 1,240	$ 94	$ -	$ 29,233
Total	K USD	$ 1,351	$ 2,791	$ 2,816	$ 2,818	$ 2,816	$ 2,816	$ 2,816	$ 2,818	$ 2,816	$ 2,816	$ 2,816	$ 2,818	$ 2,077	$ 2,077	$ 2,077	$ 2,079	$ 2,066	$ 169	$ -	$ 42,855
Consumables	$/t	$ 0.06	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.04	$ 0.07	$ 0.09	$ 0.13	$ 0.18	$ 0.26	$ 1.01	$ -	$ 0.03
Parts / MARC Cost	$/t	$ 0.01	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.01	$ 0.01	$ 0.02	$ 0.02	$ 0.03	$ 0.05	$ 0.16	$ -	$ 0.01
Maintenance Labor	$/t	$ 0.23	$ 0.08	$ 0.06	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.11	$ 0.12	$ 0.17	$ 0.24	$ 0.33	$ 0.45	$ 1.46	$ -	$ 0.07
Total	$/t	$ 0.30	$ 0.11	$ 0.08	$ 0.07	$ 0.07	$ 0.07	$ 0.08	$ 0.08	$ 0.09	$ 0.09	$ 0.09	$ 0.16	$ 0.19	$ 0.28	$ 0.39	$ 0.54	$ 0.75	$ 2.63	$ -	$ 0.11

21.5.3 Drilling

Drilling cost estimates are shown in Table 21.12. The LOM drilling costs are estimated to be $74.6 million or $0.19 per tonne including preproduction. The total LOM cost without capitalized preproduction is $73.5 million.

Table 21-12 Yearly Drilling Costs

Drilling Operating 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Prod Drill Fuel Consumption	K Liters	221	1,267	1,696	1,934	2,188	1,974	1,727	1,724	1,684	1,684	1,684	922	553	386	271	196	141	3	-	20,255
Prod Drill Fuel Cost	K USD	$ 146	$ 837	$ 1,120	$ 1,277	$ 1,445	$ 1,304	$ 1,140	$ 1,138	$ 1,112	$ 1,112	$ 1,112	$ 609	$ 365	$ 255	$ 179	$ 130	$ 93	$ 2	$ -	$ 13,377
Prod Drill Lube & Oil	K USD	$ 94	$ 537	$ 719	$ 820	$ 928	$ 837	$ 732	$ 731	$ 714	$ 714	$ 714	$ 391	$ 235	$ 164	$ 115	$ 83	$ 60	$ 1	$ -	$ 8,591
Prod Drill Undercarriage	K USD	$ 12	$ 69	$ 93	$ 106	$ 120	$ 108	$ 94	$ 94	$ 92	$ 92	$ 92	$ 50	$ 30	$ 21	$ 15	$ 11	$ 8	$ 0	$ -	$ 1,108
Prod Drill Drill Bits & Steel	K USD	$ 73	$ 420	$ 562	$ 641	$ 725	$ 655	$ 572	$ 572	$ 558	$ 558	$ 558	$ 306	$ 183	$ 128	$ 90	$ 65	$ 47	$ 1	$ -	$ 6,716
Prod Drill Total Consumables	K USD	$ 324	$ 1,863	$ 2,495	$ 2,845	$ 3,218	$ 2,904	$ 2,539	$ 2,535	$ 2,477	$ 2,477	$ 2,477	$ 1,356	$ 813	$ 568	$ 398	$ 288	$ 207	$ 5	$ -	$ 29,792
Prod Drill Parts	K USD	$ 156	$ 895	$ 1,198	$ 1,366	$ 1,546	$ 1,395	$ 1,220	$ 1,218	$ 1,190	$ 1,190	$ 1,190	$ 651	$ 391	$ 273	$ 191	$ 139	$ 99	$ 2	$ -	$ 14,308
Prod Drill Maintenance Labor	K USD	$ 170	$ 614	$ 764	$ 913	$ 928	$ 897	$ 787	$ 771	$ 755	$ 755	$ 755	$ 441	$ 330	$ 252	$ 205	$ 189	$ 189	$ 16	$ -	$ 9,730
Pioneer Drill Fuel Consumption	K Liters	7	-	76	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	82
Pioneer Drill Fuel Cost	K USD	$ 5	$ -	$ 50	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 54
Pioneer Drill Lube & Oil	K USD	$ 2	$ -	$ 21	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 23
Pioneer Drill Undercarriage	K USD	$ 1	$ -	$ 15	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 16
Pioneer Drill Drill Bits & Steel	K USD	$ 1	$ -	$ 6	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 7
Pioneer Drill Total Consumables	K USD	$ 8	$ -	$ 92	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 101
Pioneer Drill Parts / MARC Cost	K USD	$ 1	$ -	$ 6	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 7
Pioneer Drill Maintenance Labor	K USD	$ 34	$ -	$ 85	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 119
Total Drill Fuel Consumption	K Liters	227	1,267	1,772	1,934	2,188	1,974	1,727	1,724	1,684	1,684	1,684	922	553	386	271	196	141	3	-	20,337
Total Drill Fuel Cost	K USD	$ 150	$ 837	$ 1,170	$ 1,277	$ 1,445	$ 1,304	$ 1,140	$ 1,138	$ 1,112	$ 1,112	$ 1,112	$ 609	$ 365	$ 255	$ 179	$ 130	$ 93	$ 2	$ -	$ 13,431
Total Drill Lube & Oil	K USD	$ 95	$ 537	$ 741	$ 820	$ 928	$ 837	$ 732	$ 731	$ 714	$ 714	$ 714	$ 391	$ 235	$ 164	$ 115	$ 83	$ 60	$ 1	$ -	$ 8,614
Total Drill Undercarriage	K USD	$ 13	$ 69	$ 108	$ 106	$ 120	$ 108	$ 94	$ 94	$ 92	$ 92	$ 92	$ 50	$ 30	$ 21	$ 15	$ 11	$ 8	$ 0	$ -	$ 1,125
Total Drill Drill Bits & Steel	K USD	$ 74	$ 420	$ 569	$ 641	$ 725	$ 655	$ 572	$ 572	$ 558	$ 558	$ 558	$ 306	$ 183	$ 128	$ 90	$ 65	$ 47	$ 1	$ -	$ 6,723
Total Drill Total Consumables	K USD	$ 333	$ 1,863	$ 2,587	$ 2,845	$ 3,218	$ 2,904	$ 2,539	$ 2,535	$ 2,477	$ 2,477	$ 2,477	$ 1,356	$ 813	$ 568	$ 398	$ 288	$ 207	$ 5	$ -	$ 29,893
Total Drill Parts / MARC Cost	K USD	$ 156	$ 895	$ 1,205	$ 1,366	$ 1,546	$ 1,395	$ 1,220	$ 1,218	$ 1,190	$ 1,190	$ 1,190	$ 651	$ 391	$ 273	$ 191	$ 139	$ 99	$ 2	$ -	$ 14,315
Total Drill Maintenance Labor	K USD	$ 205	$ 614	$ 850	$ 913	$ 928	$ 897	$ 787	$ 771	$ 755	$ 755	$ 755	$ 441	$ 330	$ 252	$ 205	$ 189	$ 189	$ 16	$ -	$ 9,849
Total Drill Total Maintenance Allocation	K USD	$ 361	$ 1,508	$ 2,054	$ 2,279	$ 2,474	$ 2,291	$ 2,006	$ 1,989	$ 1,945	$ 1,945	$ 1,945	$ 1,092	$ 721	$ 525	$ 396	$ 327	$ 288	$ 18	$ -	$ 24,164
Total Operator Wages & Burden	K USD	$ 426	$ 1,278	$ 1,770	$ 1,901	$ 1,934	$ 1,868	$ 1,639	$ 1,606	$ 1,573	$ 1,573	$ 1,573	$ 918	$ 688	$ 524	$ 426	$ 393	$ 393	$ 33	$ -	$ 20,520
Total Drilling Cost	K USD	$ 1,120	$ 4,650	$ 6,412	$ 7,025	$ 7,626	$ 7,064	$ 6,185	$ 6,130	$ 5,996	$ 5,996	$ 5,996	$ 3,366	$ 2,223	$ 1,617	$ 1,220	$ 1,009	$ 888	$ 56	$ -	$ 74,578
Drilling Cost per Tonne Mined by Item																					
Fuel Cost	$/t	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 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	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ -	$ 0.02
Undercarriage	$/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	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ -	$ 0.00
Drill Bits & Steel	$/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	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ -	$ 0.02
Total Consumables	$/t	$ 0.07	$ 0.08	$ 0.07	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ -	$ 0.08
Parts / MARC Cost	$/t	$ 0.04	$ 0.04	$ 0.03	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ -	$ 0.04
Maintenance Labor	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.04	$ 0.05	$ 0.07	$ 0.24	$ -	$ 0.02
Total Maintenance Allocation	$/t	$ 0.08	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.07	$ 0.07	$ 0.07	$ 0.09	$ 0.11	$ 0.28	$ -	$ 0.06
Operator Wages & Burden	$/t	$ 0.10	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.07	$ 0.08	$ 0.10	$ 0.14	$ 0.51	$ -	$ 0.05
Total Drilling Cost	$/t	$ 0.25	$ 0.19	$ 0.19	$ 0.19	$ 0.18	$ 0.18	$ 0.18	$ 0.18	$ 0.18	$ 0.18	$ 0.18	$ 0.19	$ 0.21	$ 0.21	$ 0.23	$ 0.26	$ 0.32	$ 0.87	$ -	$ 0.19

21.5.4 Blasting

LOM blasting costs, including preproduction, are shown in Table 21.13. 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.

The LOM blasting costs are estimated to be $84.0 million or $0.21 per tonne including preproduction. The total LOM cost without capitalized preproduction is $82.9 million.

Table 21-13 Yearly Blasting Costs

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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Fuel	K Liters	47	93	93	93	93	93	93	93	93	93	93	93	93	93	93	93	93	8	-	1,536
Blasting Consumables	K USD	$ 868	$ 4,753	$ 6,810	$ 7,258	$ 8,209	$ 7,408	$ 6,478	$ 6,468	$ 6,319	$ 6,319	$ 6,319	$ 3,460	$ 2,075	$ 1,450	$ 1,016	$ 736	$ 527	$ 12	$ -	$ 76,487
Equipment Consumables	K USD	$ 40	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 79	$ 7	$ -	$ 1,308
Equipment Maintenance Allocations	K USD	$ 7	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 14	$ 1	$ -	$ 236
Personnel	K USD	$ 169	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 338	$ 28	$ -	$ 5,610
Supplies	K USD	$ 6	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 1	$ -	$ 199
Outside Services	K USD	$ 6	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 1	$ -	$ 199
Total Blasting Costs	K USD	$ 1,096	$ 5,208	$ 7,265	$ 7,714	$ 8,665	$ 7,863	$ 6,934	$ 6,923	$ 6,775	$ 6,775	$ 6,775	$ 3,916	$ 2,530	$ 1,905	$ 1,472	$ 1,191	$ 983	$ 50	$ -	$ 84,040
Cost per Ton																					
Blasting Consumables	$/t	$ 0.20	$ 0.19	$ 0.20	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ 0.19	$ -	$ 0.19
Equipment Consumables	$/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	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.03	$ 0.10	$ -	$ 0.00
Equipment Maintenance Allocations	$/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	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.01	$ 0.02	$ -	$ 0.00
Personnel	$/t	$ 0.04	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.03	$ 0.04	$ 0.06	$ 0.09	$ 0.12	$ 0.44	$ -	$ 0.01
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	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.02	$ -	$ 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	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.02	$ -	$ 0.00
Total	$/t	$ 0.25	$ 0.21	$ 0.21	$ 0.20	$ 0.20	$ 0.20	$ 0.21	$ 0.21	$ 0.21	$ 0.21	$ 0.21	$ 0.22	$ 0.23	$ 0.25	$ 0.28	$ 0.31	$ 0.36	$ 0.78	$ -	$ 0.21

21.5.5 Loading

Loading costs are based on operation of two hydraulic shovels with 23-cubic meter (30.0-cubic yards) buckets for all primary production. In addition, a 13-cubic meter (17.0-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 21.14.

The LOM loading costs are estimated to be $113.7 million or $0.29 per tonne including preproduction. The total LOM cost without capitalized preproduction is $112.1 million.

Table 21-14 Yearly Loading Costs

Shovel 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Fuel Consumption	K Liters	513	2,859	3,805	3,791	3,990	3,776	3,583	3,865	3,802	3,802	3,802	2,082	1,177	-	-	-	-	-	-	40,847
Fuel Cost	K USD	$ 339	$ 1,888	$ 2,513	$ 2,504	$ 2,635	$ 2,494	$ 2,367	$ 2,553	$ 2,511	$ 2,511	$ 2,511	$ 1,375	$ 777	$ -	$ -	$ -	$ -	$ -	$ -	$ 26,977
Lube & Oil	K USD	$ 125	$ 698	$ 929	$ 926	$ 974	$ 922	$ 875	$ 944	$ 928	$ 928	$ 928	$ 508	$ 287	$ -	$ -	$ -	$ -	$ -	$ -	$ 9,974
Tires / Under Carriage	K USD	$ 25	$ 141	$ 187	$ 186	$ 196	$ 186	$ 176	$ 190	$ 187	$ 187	$ 187	$ 102	$ 58	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,008
Wear Items & GET	K USD	$ 81	$ 449	$ 598	$ 596	$ 627	$ 593	$ 563	$ 608	$ 598	$ 598	$ 598	$ 327	$ 185	$ -	$ -	$ -	$ -	$ -	$ -	$ 6,421
Total Consumables	K USD	$ 570	$ 3,176	$ 4,227	$ 4,212	$ 4,433	$ 4,194	$ 3,981	$ 4,294	$ 4,223	$ 4,223	$ 4,223	$ 2,312	$ 1,308	$ -	$ -	$ -	$ -	$ -	$ -	$ 45,379
Parts / MARC Cost	K USD	$ 498	$ 2,775	$ 3,694	$ 3,680	$ 3,874	$ 3,665	$ 3,478	$ 3,752	$ 3,690	$ 3,690	$ 3,690	$ 2,021	$ 1,142	$ -	$ -	$ -	$ -	$ -	$ -	$ 39,650
Total Equip. Allocation (no labor)	K USD	$ 1,069	$ 5,952	$ 7,921	$ 7,892	$ 8,307	$ 7,859	$ 7,459	$ 8,046	$ 7,913	$ 7,913	$ 7,913	$ 4,333	$ 2,450	$ -	$ -	$ -	$ -	$ -	$ -	$ 85,029
Loader Cost																					
Fuel Consumption	K Liters	5	73	82	497	528	407	242	199	52	75	86	41	79	471	341	253	194	68	-	3,694
Fuel Cost	K USD	$ 3	$ 48	$ 54	$ 328	$ 349	$ 269	$ 160	$ 131	$ 34	$ 49	$ 57	$ 27	$ 52	$ 311	$ 225	$ 167	$ 128	$ 45	$ -	$ 2,440
Lube & Oil	K USD	$ 1	$ 15	$ 17	$ 105	$ 112	$ 86	$ 51	$ 42	$ 11	$ 16	$ 18	$ 9	$ 17	$ 100	$ 72	$ 54	$ 41	$ 14	$ -	$ 784
Tires / Under Carriage	K USD	$ 0	$ 1	$ 1	$ 6	$ 6	$ 5	$ 3	$ 2	$ 1	$ 1	$ 1	$ 0	$ 1	$ 5	$ 4	$ 3	$ 2	$ 1	$ -	$ 41
Wear Items & GET	K USD	$ 1	$ 9	$ 10	$ 60	$ 64	$ 50	$ 30	$ 24	$ 6	$ 9	$ 10	$ 5	$ 10	$ 57	$ 42	$ 31	$ 24	$ 8	$ -	$ 450
Total Consumables	K USD	$ 5	$ 73	$ 83	$ 500	$ 531	$ 410	$ 244	$ 200	$ 53	$ 75	$ 86	$ 41	$ 80	$ 473	$ 343	$ 255	$ 195	$ 69	$ -	$ 3,715
Parts / MARC Cost	K USD	$ 2	$ 24	$ 27	$ 162	$ 173	$ 133	$ 79	$ 65	$ 17	$ 24	$ 28	$ 13	$ 26	$ 154	$ 112	$ 83	$ 63	$ 22	$ -	$ 1,208
Total Equip. Allocation (no labor)	K USD	$ 6	$ 97	$ 109	$ 662	$ 704	$ 543	$ 323	$ 265	$ 70	$ 100	$ 114	$ 55	$ 106	$ 627	$ 455	$ 338	$ 258	$ 91	$ -	$ 4,922
Total Loading Cost																					
Fuel Consumption	K Liters	518	2,932	3,887	4,288	4,519	4,183	3,826	4,064	3,854	3,876	3,887	2,123	1,256	471	341	253	194	68	-	44,541
Fuel Cost	K USD	$ 342	$ 1,936	$ 2,567	$ 2,832	$ 2,984	$ 2,763	$ 2,527	$ 2,684	$ 2,545	$ 2,560	$ 2,567	$ 1,402	$ 830	$ 311	$ 225	$ 167	$ 128	$ 45	$ -	$ 29,416
Lube & Oil	K USD	$ 126	$ 714	$ 947	$ 1,031	$ 1,086	$ 1,008	$ 926	$ 986	$ 939	$ 944	$ 946	$ 517	$ 304	$ 100	$ 72	$ 54	$ 41	$ 14	$ -	$ 10,757
Tires / Under Carriage	K USD	$ 25	$ 141	$ 188	$ 192	$ 202	$ 190	$ 179	$ 192	$ 187	$ 188	$ 188	$ 103	$ 59	$ 5	$ 4	$ 3	$ 2	$ 1	$ -	$ 2,049
Wear Items & GET	K USD	$ 81	$ 458	$ 608	$ 656	$ 692	$ 643	$ 593	$ 632	$ 604	$ 607	$ 608	$ 332	$ 195	$ 57	$ 42	$ 31	$ 24	$ 8	$ -	$ 6,871
Total Consumables	K USD	$ 575	$ 3,249	$ 4,310	$ 4,712	$ 4,965	$ 4,604	$ 4,225	$ 4,494	$ 4,276	$ 4,299	$ 4,309	$ 2,354	$ 1,387	$ 473	$ 343	$ 255	$ 195	$ 69	$ -	$ 49,093
Parts / MARC Cost	K USD	$ 500	$ 2,799	$ 3,721	$ 3,843	$ 4,046	$ 3,798	$ 3,558	$ 3,817	$ 3,707	$ 3,715	$ 3,718	$ 2,034	$ 1,168	$ 154	$ 112	$ 83	$ 63	$ 22	$ -	$ 40,858
Total Equip. Allocation (no labor)	K USD	$ 1,075	$ 6,049	$ 8,030	$ 8,554	$ 9,011	$ 8,402	$ 7,782	$ 8,311	$ 7,983	$ 8,013	$ 8,028	$ 4,388	$ 2,556	$ 627	$ 455	$ 338	$ 258	$ 91	$ -	$ 89,952
Maintenance Labor	K USD	$ 205	$ 614	$ 850	$ 913	$ 928	$ 897	$ 787	$ 771	$ 755	$ 755	$ 755	$ 441	$ 330	$ 252	$ 205	$ 189	$ 189	$ 16	$ -	$ 9,849
Operator Wages & Burden	K USD	$ 275	$ 1,032	$ 1,187	$ 1,239	$ 1,239	$ 1,204	$ 1,170	$ 1,153	$ 1,032	$ 1,032	$ 1,032	$ 809	$ 533	$ 206	$ 206	$ 206	$ 206	$ 120	$ -	$ 13,886
Total Loading Costs	K USD	$ 1,555	$ 7,695	$ 10,067	$ 10,706	$ 11,178	$ 10,504	$ 9,739	$ 10,235	$ 9,771	$ 9,801	$ 9,815	$ 5,637	$ 3,420	$ 1,086	$ 866	$ 733	$ 653	$ 227	$ -	$ 113,687
Cost per Ton																					
Fuel Cost	$/t	$ 0.08	$ 0.08	$ 0.07	$ 0.08	$ 0.07	$ 0.07	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.04	$ 0.04	$ 0.04	$ 0.05	$ 0.70	$ -	$ 0.07
Lube & Oil	$/t	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.22	$ -	$ 0.03
Tires / Under Carriage	$/t	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.00	$ 0.00	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 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	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.13	$ -	$ 0.02
Total Consumables	$/t	$ 0.13	$ 0.13	$ 0.12	$ 0.12	$ 0.12	$ 0.12	$ 0.13	$ 0.13	$ 0.13	$ 0.13	$ 0.13	$ 0.13	$ 0.13	$ 0.06	$ 0.06	$ 0.07	$ 0.07	$ 1.07	$ -	$ 0.12
Parts / MARC Cost	$/t	$ 0.11	$ 0.11	$ 0.11	$ 0.10	$ 0.09	$ 0.10	$ 0.11	$ 0.11	$ 0.11	$ 0.11	$ 0.11	$ 0.11	$ 0.11	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.35	$ -	$ 0.10
Total Equip. Allocation (no labor)	$/t	$ 0.24	$ 0.24	$ 0.23	$ 0.23	$ 0.21	$ 0.22	$ 0.23	$ 0.25	$ 0.24	$ 0.24	$ 0.24	$ 0.24	$ 0.24	$ 0.08	$ 0.09	$ 0.09	$ 0.09	$ 1.41	$ -	$ 0.23
Maintenance Labor	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.04	$ 0.05	$ 0.07	$ 0.24	$ -	$ 0.02
Operator Wages & Burden	$/t	$ 0.06	$ 0.04	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.05	$ 0.03	$ 0.04	$ 0.05	$ 0.08	$ 1.87	$ -	$ 0.04
Total Loading Cost	$/t	$ 0.35	$ 0.31	$ 0.29	$ 0.28	$ 0.26	$ 0.27	$ 0.29	$ 0.30	$ 0.30	$ 0.30	$ 0.30	$ 0.31	$ 0.32	$ 0.14	$ 0.16	$ 0.19	$ 0.24	$ 3.53	$ -	$ 0.29

21.5.6 Hauling

Haulage cost was estimated using the truck hour estimates discussed in Section 16.3. The yearly haulage cost estimate is shown in Table 21.15. This includes the use of Railveyor to lower the cost of ore haulage. Of note, the Railveyor cost is reduced for Florida Mountain as it regenerates more power than it consumes.

The LOM haulage costs are estimated to be $301.9 million or $0.76 per tonne including preproduction. The total LOM cost without capitalized preproduction is $297.7 million.

Table 21-15 Yearly Haulage Costs

Haulage 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Fuel Consumption	K Liters	1,533	7,795	10,843	15,083	12,678	13,900	11,168	9,331	7,383	8,897	9,217	5,797	3,909	3,074	2,279	1,716	1,310	207	-	126,118
Fuel Cost	K USD	$ 1,012	$ 5,148	$ 7,161	$ 9,961	$ 8,373	$ 9,180	$ 7,376	$ 6,162	$ 4,876	$ 5,876	$ 6,087	$ 3,829	$ 2,582	$ 2,030	$ 1,505	$ 1,133	$ 865	$ 137	$ -	$ 83,292
Lube & Oil	K USD	$ 451	$ 2,296	$ 3,193	$ 4,442	$ 3,733	$ 4,093	$ 3,289	$ 2,748	$ 2,174	$ 2,620	$ 2,714	$ 1,707	$ 1,151	$ 905	$ 671	$ 505	$ 386	$ 61	$ -	$ 37,140
Tires	K USD	$ 517	$ 2,630	$ 3,658	$ 5,088	$ 4,276	$ 4,689	$ 3,767	$ 3,147	$ 2,490	$ 3,001	$ 3,109	$ 1,956	$ 1,319	$ 1,037	$ 769	$ 579	$ 442	$ 70	$ -	$ 42,543
Wear Items & GET	K USD	$ 31	$ 159	$ 221	$ 307	$ 258	$ 283	$ 227	$ 190	$ 150	$ 181	$ 188	$ 118	$ 80	$ 63	$ 46	$ 35	$ 27	$ 4	$ -	$ 2,567
Total Consumables	K USD	$ 2,012	$ 10,232	$ 14,233	$ 19,798	$ 16,641	$ 18,245	$ 14,659	$ 12,247	$ 9,691	$ 11,678	$ 12,098	$ 7,609	$ 5,131	$ 4,035	$ 2,991	$ 2,252	$ 1,720	$ 272	$ -	$ 165,542
Parts / MARC Cost	K USD	$ 416	$ 2,119	$ 2,947	$ 4,099	$ 3,445	$ 3,778	$ 3,035	$ 2,536	$ 2,006	$ 2,418	$ 2,505	$ 1,576	$ 1,062	$ 835	$ 619	$ 466	$ 356	$ 56	$ -	$ 34,275
Total Equip. Allocation (no labor)	K USD	$ 2,428	$ 12,351	$ 17,179	$ 23,897	$ 20,086	$ 22,023	$ 17,694	$ 14,783	$ 11,697	$ 14,095	$ 14,603	$ 9,185	$ 6,194	$ 4,871	$ 3,610	$ 2,718	$ 2,076	$ 328	$ -	$ 199,817
Maintenance Labor	K USD	$ 362	$ 1,605	$ 2,502	$ 2,832	$ 2,832	$ 2,832	$ 2,533	$ 2,077	$ 2,077	$ 2,077	$ 2,077	$ 1,731	$ 1,133	$ 771	$ 566	$ 566	$ 535	$ 220	$ -	$ 29,328
Operator Wages & Burden	K USD	$ 724	$ 3,210	$ 5,003	$ 5,979	$ 6,042	$ 6,042	$ 5,129	$ 4,154	$ 4,154	$ 4,154	$ 4,154	$ 3,461	$ 2,266	$ 1,542	$ 1,133	$ 1,133	$ 1,070	$ 441	$ -	$ 59,788
Total Haulage Costs	K USD	$ 3,514	$ 17,165	$ 24,684	$ 32,708	$ 28,960	$ 30,896	$ 25,356	$ 21,014	$ 17,927	$ 20,326	$ 20,833	$ 14,377	$ 9,592	$ 7,184	$ 5,310	$ 4,418	$ 3,681	$ 989	$ -	$ 288,933
Rail Veyor Costs																					
Power Consumption	mWh	302	(2,300)	(2,300)	(2,383)	(2,371)	2,745	6,924	6,143	4,179	1,627	1,177	1,439	13,330	1,328	1,322	1,353	1,293	602	-	34,411
Power Cost	K USD	20	(149)	(149)	(155)	(154)	178	450	399	272	106	76	94	866	86	86	88	84	39	-	2,237
Maintenance Cost	K USD	$ 591	$ 655	$ 655	$ 450	$ 636	$ 1,580	$ 758	$ 893	$ 585	$ 424	$ 396	$ 413	$ 406	$ 593	$ 405	$ 487	$ 403	$ 360	$ -	$ 10,692
Total Rail Veyor Cost	K USD	$ 611	$ 505	$ 505	$ 295	$ 482	$ 1,759	$ 1,208	$ 1,293	$ 857	$ 530	$ 473	$ 506	$ 1,272	$ 679	$ 491	$ 575	$ 487	$ 399	$ -	$ 12,929
Total Haulage Costs	K USD	$ 4,124	$ 17,670	$ 25,190	$ 33,003	$ 29,442	$ 32,655	$ 26,565	$ 22,306	$ 18,784	$ 20,856	$ 21,306	$ 14,883	$ 10,864	$ 7,863	$ 5,801	$ 4,993	$ 4,169	$ 1,388	$ -	$ 301,862
Cost per Tonne Moved																					
Fuel Cost	$/t	$ 0.23	$ 0.21	$ 0.21	$ 0.26	$ 0.20	$ 0.24	$ 0.22	$ 0.18	$ 0.15	$ 0.18	$ 0.19	$ 0.21	$ 0.24	$ 0.27	$ 0.28	$ 0.30	$ 0.32	$ 2.13	$ -	$ 0.21
Lube & Oil	$/t	$ 0.10	$ 0.09	$ 0.09	$ 0.12	$ 0.09	$ 0.11	$ 0.10	$ 0.08	$ 0.07	$ 0.08	$ 0.08	$ 0.09	$ 0.11	$ 0.12	$ 0.13	$ 0.13	$ 0.14	$ 0.95	$ -	$ 0.09
Tires	$/t	$ 0.12	$ 0.11	$ 0.11	$ 0.13	$ 0.10	$ 0.12	$ 0.11	$ 0.09	$ 0.08	$ 0.09	$ 0.09	$ 0.11	$ 0.12	$ 0.14	$ 0.15	$ 0.15	$ 0.16	$ 1.09	$ -	$ 0.11
Wear Items & GET	$/t	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.00	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.07	$ -	$ 0.01
Total Consumables	$/t	$ 0.45	$ 0.41	$ 0.41	$ 0.52	$ 0.39	$ 0.47	$ 0.44	$ 0.36	$ 0.29	$ 0.36	$ 0.37	$ 0.42	$ 0.48	$ 0.54	$ 0.57	$ 0.59	$ 0.63	$ 4.22	$ -	$ 0.42
Parts / MARC Cost	$/t	$ 0.09	$ 0.09	$ 0.09	$ 0.11	$ 0.08	$ 0.10	$ 0.09	$ 0.08	$ 0.06	$ 0.07	$ 0.08	$ 0.09	$ 0.10	$ 0.11	$ 0.12	$ 0.12	$ 0.13	$ 0.87	$ -	$ 0.09
Total Equip. Allocation (no labor)	$/t	$ 0.55	$ 0.50	$ 0.50	$ 0.63	$ 0.47	$ 0.57	$ 0.53	$ 0.44	$ 0.36	$ 0.43	$ 0.44	$ 0.51	$ 0.57	$ 0.65	$ 0.68	$ 0.71	$ 0.76	$ 5.10	$ -	$ 0.50
Maintenance Labor	$/t	$ 0.08	$ 0.06	$ 0.07	$ 0.08	$ 0.07	$ 0.07	$ 0.08	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.10	$ 0.11	$ 0.10	$ 0.11	$ 0.15	$ 0.20	$ 3.43	$ -	$ 0.07
Operator Wages & Burden	$/t	$ 0.16	$ 0.13	$ 0.14	$ 0.16	$ 0.14	$ 0.16	$ 0.15	$ 0.12	$ 0.13	$ 0.13	$ 0.13	$ 0.19	$ 0.21	$ 0.20	$ 0.21	$ 0.30	$ 0.39	$ 6.85	$ -	$ 0.15
Total Haulage Costs	$/t	$ 0.79	$ 0.69	$ 0.71	$ 0.87	$ 0.68	$ 0.80	$ 0.75	$ 0.63	$ 0.55	$ 0.62	$ 0.63	$ 0.80	$ 0.89	$ 0.95	$ 1.01	$ 1.15	$ 1.34	$ 15.38	$ -	$ 0.73
Rail Veyor Costs																					
Power Cost	$/t	$ 0.00	$ (0.01)	$ (0.00)	$ (0.00)	$ (0.00)	$ 0.00	$ 0.01	$ 0.01	$ 0.01	$ 0.00	$ 0.00	$ 0.01	$ 0.08	$ 0.01	$ 0.02	$ 0.02	$ 0.03	$ 0.61	$ -	$ 0.01
Maintenance Cost	$/t	$ 0.13	$ 0.03	$ 0.02	$ 0.01	$ 0.01	$ 0.04	$ 0.02	$ 0.03	$ 0.02	$ 0.01	$ 0.01	$ 0.02	$ 0.04	$ 0.08	$ 0.08	$ 0.13	$ 0.15	$ 5.60	$ -	$ 0.03
Total Rail Veyor Cost	$/t	$ 0.14	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.05	$ 0.04	$ 0.04	$ 0.03	$ 0.02	$ 0.01	$ 0.03	$ 0.12	$ 0.09	$ 0.09	$ 0.15	$ 0.18	$ 6.21	$ -	$ 0.03
Total Haulage Costs	$/t	$ 0.93	$ 0.72	$ 0.73	$ 0.87	$ 0.69	$ 0.85	$ 0.79	$ 0.66	$ 0.57	$ 0.63	$ 0.65	$ 0.83	$ 1.01	$ 1.04	$ 1.10	$ 1.31	$ 1.52	$ 21.58	$ -	$ 0.76

21.5.7 Mine Support

Yearly mine support cost estimates are shown in Table 21.16 including preproduction costs. These costs assume the hourly costs for required support equipment and personnel as discussed in Sections 16.3 and 16.4 respectively.

The LOM support costs are estimated to be $85.0 million or $0.21 per tonne including preproduction. The total LOM cost without capitalized preproduction is $82.8 million.

Table 21-16 Yearly Mine Support Costs

Total 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	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Consumables	K USD	$ 887	$ 2,651	$ 2,651	$ 2,659	$ 2,651	$ 2,651	$ 2,651	$ 2,659	$ 2,651	$ 2,651	$ 2,651	$ 1,774	$ 1,477	$ 1,477	$ 1,477	$ 1,481	$ 1,477	$ 125	$ -	$ 36,704
Parts / MARC Cost	K USD	$ 396	$ 1,048	$ 1,048	$ 1,050	$ 1,048	$ 1,048	$ 1,048	$ 1,050	$ 1,048	$ 1,048	$ 1,048	$ 692	$ 572	$ 572	$ 572	$ 573	$ 572	$ 49	$ -	$ 14,479
Maintenance Labor	K USD	$ 283	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 802	$ 555	$ 472	$ 472	$ 472	$ 472	$ 472	$ 39	$ -	$ 11,261
Operating Labor	K USD	$ 566	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,605	$ 1,109	$ 944	$ 944	$ 944	$ 944	$ 944	$ 79	$ -	$ 22,523
Total	K USD	$ 2,133	$ 6,106	$ 6,106	$ 6,116	$ 6,106	$ 6,106	$ 6,106	$ 6,116	$ 6,106	$ 6,106	$ 6,106	$ 4,130	$ 3,465	$ 3,465	$ 3,465	$ 3,470	$ 3,465	$ 292	$ -	$ 84,967
Cost per Tonne Mined																					
Consumables	$/t	$ 0.20	$ 0.11	$ 0.08	$ 0.07	$ 0.06	$ 0.07	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.08	$ 0.10	$ 0.14	$ 0.20	$ 0.28	$ 0.39	$ 0.54	$ 1.95	$ -	$ 0.09
Maintenance Allocations	$/t	$ 0.09	$ 0.04	$ 0.03	$ 0.03	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.05	$ 0.08	$ 0.11	$ 0.15	$ 0.21	$ 0.76	$ -	$ 0.04
Maintenance Labor	$/t	$ 0.06	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.04	$ 0.06	$ 0.09	$ 0.12	$ 0.17	$ 0.61	$ -	$ 0.03
Operating Labor	$/t	$ 0.13	$ 0.06	$ 0.05	$ 0.04	$ 0.04	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.09	$ 0.13	$ 0.18	$ 0.25	$ 0.34	$ 1.22	$ -	$ 0.06
Total Costs	$/t	$ 0.48	$ 0.25	$ 0.18	$ 0.16	$ 0.14	$ 0.16	$ 0.18	$ 0.18	$ 0.19	$ 0.19	$ 0.19	$ 0.23	$ 0.32	$ 0.46	$ 0.66	$ 0.91	$ 1.26	$ 4.54	$ -	$ 0.21

21.6 Process Operating Cost Summary

Process operating costs were developed by M3 Engineering from first principles. Labor costs were estimated using project specific staffing, salary and wage, and benefit requirements. Unit consumptions of materials, supplies, power, and delivered supply costs were also estimated. The consumptions of cyanide and lime vary between the ore domains. These are applied in the financial model, outside of the static cost per tonne, to allow for the variations in reagent consumptions to be allocated according to the source of the ore. Table 21.17 lists the average operating costs per tonne at full plant capacity for the oxide and mixed heap-leach plant and for the non-oxide plant. As aforementioned, the individual costs per tonne vary in the financial model to apply the cyanide and lime consumptions per domain. Furthermore, the process operating overall costs per year are also dependent on quantity of each ore type as well as the total power consumptions.

Table 21-17 Operating Costs Ratios for Deposit/Ore Types

Deposit	DeLamar	DeLamar	Florida Mtn	Florida Mtn	DeLamar	Florida Mtn
Ore Type	Oxide	Mixed	Oxide	Mixed	Non-Oxide	Non-Oxide
Process Plant Capacity (tpd)	**35,000**	**35,000**	**35,000**	**35,000**	**6,000**	**6,000**
Operating Labor ($/t)	$0.27	$0.27	$0.27	$0.27	$1.65	$1.65
Power ($/t)	$0.35	$0.35	$0.35	$0.35	$1.74	$1.74
Consumables ($/t)	$1.82	$2.51	$1.43	$2.71	$7.48	$5.29
Maintenance ($/t)	$0.27	$0.27	$0.27	$0.27	$1.38	$1.38
Supplies & Services ($/t)	$0.08	$0.08	$0.08	$0.08	$0.55	$0.55
Total Process Plant ($/t)	**$2.78**	**$3.47**	**$2.39**	**$3.67**	**$12.80**	**$10.60**

Operating costs were estimated based on 4th quarter 2021 US dollars and are presented with no added contingency based upon the design and operating criteria present in this Technical Report. Operating costs are considered to have an accuracy of +/- 20%.

The process operating costs presented are based upon the ownership of all process production equipment and site facilities. The owner will employ and direct all operating maintenance and support personnel for all site activities.

Operating costs estimates have been based upon information obtained from the following sources:

- Project metallurgical test work and process engineering;
- Development of a detailed equipment list and demand/consumption calculations;
- M3 Engineering in-house data for reagent pricing; and
- Experience with other similar operations.

Where specific data does not exist, cost allowances have been based upon consumption and operating requirements from other similar properties for which reliable data exist.

21.6.1 Power

Power usage for the process and process facilities was derived from estimated connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation were assigned and coupled with estimated on-stream times to determine the average energy usage and cost. Power requirements for the heap-leach and Merrill-Crowe operation are presented in Table 21.18. Power requirements for the non-oxide mill operation are presented in Table 21.19.

Table 21-18 Power Summary for Heap Leach & Merrill Crowe Facility

Area Description	Connected Power (kW)	Demand Power (kW)	Consumed Power (kW)	Annual Consumption (kWh)
Primary Crushing	1,607	1,406	865	7,469,439
Fine Crushing	5,798	5,146	3,520	30,413,698
Agglomeration	436	389	266	2,302,337
Conveying/Stacking	3,158	2,691	1,684	14,552,257
Heap Leach Solution Systems	6,867	1,787	1,730	14,943,253
Merrill Crowe	1,795	1,466	1,436	12,410,402
Refinery	368	198	194	1,673,799
Water Systems	795	406	398	3,437,362
Reagents	25	8	8	71,288
Compressed Air	57	26	26	221,710
Total Process Plant	**20,907**	**13,524**	**10,127**	**87,495,544**

Table 21-19 Power Summary for Non-Oxide Mill Facility

Area Description	Connected Power (kW)	Demand Power (kW)	Consumed Power (kW)	Annual Consumption (kWh)
Primary Crushed Ore Conveyance	1,141	171	103	891,888
Crushed Ore Reclaim	204	168	134	1,159,514
Grinding	7,094	6,324	5,518	47,673,955
Flotation & Concentrate Regrind	2,309	2,034	1,871	16,167,642
Concentrate Leaching	259	210	191	1,652,903
Concentrate CCD	419	305	277	2,391,342
Flotation Tailing Thickening	498	258	235	2,028,972
Water Systems	297	252	232	2,002,910
Reagents	252	146	135	1,163,454
Total Process Plant	**12,473**	**9,869**	**8,696**	**75,132,581**

21.6.2 Consumable Items

Operating supplies have been estimated based upon unit costs and consumption rates projected by metallurgical tests, which are detailed in Section 13 of this report. Freight costs are included in all operating supply and reagent estimates. Reagent consumptions have been derived from test work and from design criteria considerations. Other consumable items have been estimated by M3 Engineering based on experience with other similar operations.

21.6.3 Maintenance

Annual maintenance costs have been included for the process facilities. The maintenance costs are estimated from the capital cost of the plant equipment at an allowance of 5% for parts repair or replacement. Maintenance labor is also included. The maintenance labor for the heap-leach and Merrill-Crowe operation includes one maintenance supervisor, four mechanics, and two electricians. These personnel are included as part of the overall process personnel quantity. The maintenance labor for the non-oxide mill operation includes one maintenance supervisor, eight mechanics, and two electricians. These personnel are included as part of the overall process personnel quantity. An allowance for outside repairs is also included at 10% of the maintenance parts allowance.

21.6.4 Supplies and Services

Estimates for supplies and services have been included for items such as lubricants, third-party services for the process plant, safety items, and minor supplies and tools outside of maintenance.

21.6.5 Application of Operating Costs

The process costs were developed by M3 Engineering based on the nominal processing rates. Mr. Dyer created the production schedule using the nominal processing rates as production targets, but due to mining in different areas, the rates were not always met. Thus, the process costs needed to be adjusted for periods where material will not be available to maximize processing rates. Mr. Dyer worked with M3 Engineering to determine which cost components to consider as fixed costs and which components were to be used as variable costs.

The fixed costs are those costs that remain relatively constant by time rather than changing based on the amount of tonnage being processed per tonne. The fixed costs considered labor, maintenance, and supplies and services as fixed costs which were applied monthly. These costs were developed using the $/t cost calculated by M3 Engineering and multiplying them against the nominal tonnage to be processed in each monthly period.

Later in the mine schedule, starting around year 9, the amount of leach material reporting to the plant will be substantially reduced. This would mean that the labor, maintenance, and supplies and services will also be significantly reduced, and though they will not become non-existent, they will have some cost to them. For this reason, once the amount of leach material reporting to the heap-leach plant becomes reduced to below 50% of the nominal rate (triggered in year 9), then the fixed costs will be reduced by 90%.

The variable costs are those that vary based on the tonnage processed. Costs for power and consumables were considered variable costs and were applied using a rate per tonne processed as provided by M3 Engineering.

The modified process operating costs used in the economic model are summarized by year in Table 21.20. The resulting costs exceed the overall cost per tonne estimated by M3 Engineering slightly due to the additional application of the fixed costs.

Mr. Dyer also modified M3 Engineering's power costs. Based on studies for power costing, M3 Engineering was given an original cost of power of $0.05/kWh. Prior to completion of the PFS, the cost of power was revised to 0.065/kWh. This would represent the realized weighted power cost based on LNG and solar generation. The result is an increase of $0.101 cost per tonne of heap leach ore and $0.521 cost per tonne of mill ore due to the revised power cost.

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 21.21 shows the yearly G&A cost estimate.

Note that preproduction costs are capitalized as part of the owner's costs (see Section 21.3.2). The resulting LOM G&A cost is $110.8 million. The cost after capitalization of preproduction costs is $105.7 million.

Table 21-20 Modified Process Operating Costs

Heap Leach Operating Cost	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Labor	K USD	$ 56	$ 2,856	$ 3,360	$ 3,360	$ 3,360	$ 3,360	$ 3,360	$ 3,108	$ 2,352	$ 336	$ 308	$ 336	$ 308	$ 280	$ 308	$ 336	$ 308	$ 28	$ -	$ 27,722
Power	K USD	$ 275	$ 4,643	$ 5,664	$ 5,703	$ 4,997	$ 5,687	$ 5,687	$ 4,926	$ 2,855	$ 450	$ 134	$ 203	$ 117	$ 80	$ 77	$ 90	$ 46	$ 0	$ -	$ 41,634
Consumables	K USD	$ 1,440	$ 26,741	$ 31,903	$ 32,087	$ 29,331	$ 32,755	$ 32,031	$ 27,994	$ 17,691	$ 2,811	$ 833	$ 1,268	$ 730	$ 497	$ 480	$ 564	$ 290	$ 0	$ -	$ 239,447
Maintenance	K USD	$ 57	$ 2,883	$ 3,392	$ 3,392	$ 3,392	$ 3,392	$ 3,392	$ 3,138	$ 2,375	$ 339	$ 311	$ 339	$ 311	$ 283	$ 311	$ 339	$ 311	$ 28	$ -	$ 27,985
Supplies & Services	K USD	$ 16	$ 810	$ 953	$ 953	$ 953	$ 953	$ 953	$ 881	$ 667	$ 95	$ 87	$ 95	$ 87	$ 79	$ 87	$ 95	$ 87	$ 8	$ -	$ 7,858
Total Heap Leach Costs	K USD	$ 1,843	$ 37,933	$ 45,272	$ 45,495	$ 42,034	$ 46,147	$ 45,423	$ 40,047	$ 25,940	$ 4,032	$ 1,673	$ 2,242	$ 1,553	$ 1,219	$ 1,263	$ 1,424	$ 1,042	$ 64	$ -	$ 344,648
Total Heap Leach Costs	$/t	$ 3.02	$ 3.69	$ 3.61	$ 3.60	$ 3.80	$ 3.66	$ 3.61	$ 3.67	$ 4.10	$ 4.04	$ 5.65	$ 4.98	$ 6.01	$ 6.92	$ 7.43	$ 7.13	$ 10.15	$ 7,864.19	$ -	$ 3.74
Non Oxide Mill Costs																					
Labor	K USD	$ -	$ -	$ -	$ 3,257	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 3,554	$ 2,073	$ -	$ 51,526
Power	K USD	$ -	$ -	$ -	$ 4,482	$ 4,884	$ 4,884	$ 4,884	$ 4,897	$ 4,884	$ 4,884	$ 4,884	$ 4,897	$ 4,884	$ 4,884	$ 4,884	$ 4,897	$ 4,884	$ 2,628	$ -	$ 70,637
Consumables	K USD	$ -	$ -	$ -	$ 11,707	$ 11,796	$ 11,992	$ 14,911	$ 14,050	$ 15,064	$ 14,740	$ 14,586	$ 15,618	$ 15,499	$ 15,679	$ 15,664	$ 15,777	$ 15,675	$ 7,305	$ -	$ 210,065
Maintenance	K USD	$ -	$ -	$ -	$ 2,730	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 2,978	$ 1,737	$ -	$ 43,187
Supplies & Services	K USD	$ -	$ -	$ -	$ 1,086	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 1,185	$ 691	$ -	$ 17,183
Total Florida Mnt Mill Costs	K USD	$ -	$ -	$ -	$ 23,263	$ 24,397	$ 24,592	$ 27,512	$ 26,664	$ 27,664	$ 27,341	$ 27,187	$ 28,232	$ 28,100	$ 28,280	$ 28,265	$ 28,391	$ 28,276	$ 14,435	$ -	$ 392,598
Total Florida Mnt Mill Costs	$/t	$ -	$ -	$ -	$ 11.73	$ 11.29	$ 11.39	$ 12.74	$ 12.31	$ 12.81	$ 12.66	$ 12.59	$ 13.03	$ 13.01	$ 13.09	$ 13.09	$ 13.11	$ 13.09	$ 12.42	$ -	$ 12.57
Total Project Processing Cost																					
Labor	K USD	$ 56	$ 2,856	$ 3,360	$ 6,618	$ 6,914	$ 6,914	$ 6,914	$ 6,662	$ 5,906	$ 3,890	$ 3,862	$ 3,890	$ 3,862	$ 3,834	$ 3,862	$ 3,890	$ 3,862	$ 2,101	$ -	$ 79,248
Power	K USD	$ 275	$ 4,643	$ 5,664	$ 10,185	$ 9,881	$ 10,571	$ 10,571	$ 9,823	$ 7,738	$ 5,334	$ 5,017	$ 5,100	$ 5,000	$ 4,963	$ 4,960	$ 4,987	$ 4,930	$ 2,628	$ -	$ 112,272
Consumables	K USD	$ 1,440	$ 26,741	$ 31,903	$ 43,794	$ 41,128	$ 44,746	$ 46,943	$ 42,044	$ 32,755	$ 17,551	$ 15,419	$ 16,886	$ 16,229	$ 16,176	$ 16,144	$ 16,341	$ 15,965	$ 7,305	$ -	$ 449,513
Maintenance	K USD	$ 57	$ 2,883	$ 3,392	$ 6,122	$ 6,371	$ 6,371	$ 6,371	$ 6,116	$ 5,353	$ 3,318	$ 3,289	$ 3,318	$ 3,289	$ 3,261	$ 3,289	$ 3,318	$ 3,289	$ 1,766	$ -	$ 71,172
Supplies & Services	K USD	$ 16	$ 810	$ 953	$ 2,039	$ 2,138	$ 2,138	$ 2,138	$ 2,066	$ 1,852	$ 1,280	$ 1,272	$ 1,280	$ 1,272	$ 1,264	$ 1,272	$ 1,280	$ 1,272	$ 699	$ -	$ 25,041
Total Process operating Costs	K USD	$ 1,843	$ 37,933	$ 45,272	$ 68,758	$ 66,431	$ 70,739	$ 72,935	$ 66,711	$ 53,604	$ 31,372	$ 28,860	$ 30,474	$ 29,653	$ 29,498	$ 29,527	$ 29,816	$ 29,318	$ 14,499	$ -	$ 737,245
Total Process operating Costs	$/t	$ 3.02	$ 3.69	$ 3.61	$ 4.70	$ 5.02	$ 4.79	$ 4.94	$ 5.10	$ 6.32	$ 9.94	$ 11.75	$ 11.65	$ 12.26	$ 12.63	$ 12.67	$ 12.60	$ 12.96	$ 12.47	$ -	$ 5.97

22.0 ECONOMIC ANALYSIS

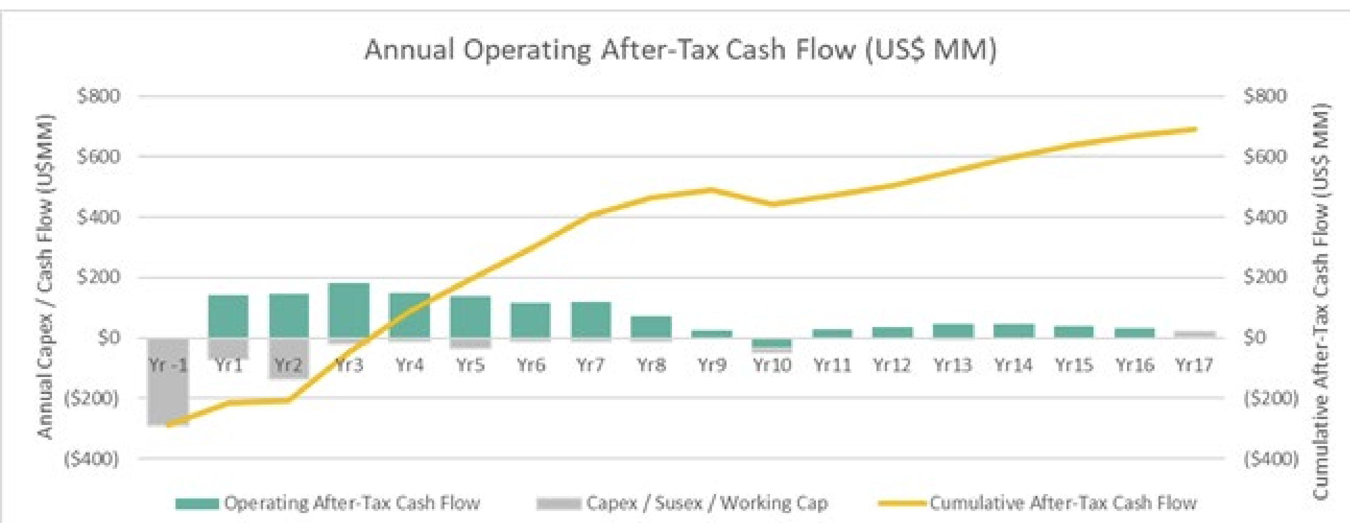
RESPEC has prepared the PFS for the DeLamar mining project, which includes operations at both DeLamar and Florida Mountain. The economic analysis uses principal assumptions which include the metal recoveries as discussed in Section 13, processing methods discussed in Section 17, metal prices discussed in Section 19, and the capital and operating costs from Section 21.

The project physicals, pre-tax cash flows, and after-tax cash flow are discussed in Sections 22.1, 22.2, and 23.3 respectively. Some economic highlights include:

- Initial construction period is anticipated to be 18 months;
- After-tax net present value (“NPV”) (5%) of $407.8 million with a 27% after-tax internal rate of return (“IRR”) using $1,700 and $21.50 per ounce gold and silver prices, respectively;
- After-tax payback period of 3.34 years;
- Year 1 to 8 gold equivalent average production of 163,000 ounces (average 121,000 oz Au/year and 3,312,000 oz Ag/year);
- Year 1 to 16 gold equivalent average production of 110,000 ounces (average 71,000 oz Au/year and 3,085,000 oz Ag/year).
- After-tax LOM cumulative cash flow of $689.3 million; and
- Average annual after-tax free cash flow of $59.8 million during production.

Figure 22.1 shows the annual operating after-tax cash flow.

Figure 22-1 Annual Operating After-Tax Cash Flow



22.1 Mining Physicals

The cash-flow model uses the mining and process production schedule physicals summarized in Table 22.1. and as discussed in Section 16.2. Gold and silver ounce production is also shown in Table 22.1 based on the metal production model provided by Mr. Botz.

The payable metal was compiled and shown in the revenue section of the cash-flow mode in Table 22.2. This assumes a 99.5% payable factor applied to the metal production in Table 22.1. The payable metal production is also shown by process method in Table 22.2 and Figure 22.3 shows the metal profile by gold and silver.

Of note: the LOM average recovery for Florida Mountain leach material is 76% and 47% for gold and silver, respectively. The LOM average DeLamar recovery for gold and silver is 66% and 32%, respectively. For mill ore, the LOM averages are 83% and 72% for Florida Mountain gold and silver, respectively. DeLamar non-oxide LOM average mill recoveries are 37% and 75% for gold and silver respectively. These recovery numbers are prior to the application of the 99.5% payable metal factor. With the payable factor, the overall LOM payable recoveries are 64% and 54% for gold and silver, respectively.

Table 22-1 Yearly Mine & Process Physicals

Material Mined	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Florida Mnt Above COG	K Tonnes	946	11,209	12,762	10,204	14,516	6,587	39	-	-	-	-	-	-	-	-	-	-	-	-	56,263
	g Au/t	0.35	0.53	0.42	0.43	0.44	0.43	0.32	-	-	-	-	-	-	-	-	-	-	-	-	0.45
	K Ozs Au	11	192	174	140	205	92	0	-	-	-	-	-	-	-	-	-	-	-	-	815
	g Ag/t	8.89	13.00	9.95	8.60	12.27	15.58	20.31	-	-	-	-	-	-	-	-	-	-	-	-	11.56
	K Ozs Ag	271	4,686	4,082	2,822	5,726	3,300	25	-	-	-	-	-	-	-	-	-	-	-	-	20,911
Florida Mnt Waste	K Tonnes	3,491	13,498	15,459	20,710	27,912	12,933	40	-	-	-	-	-	-	-	-	-	-	-	-	94,043
Florida Mnt Total Mined	K Tonnes	4,437	24,707	28,221	30,914	42,429	19,520	79	-	-	-	-	-	-	-	-	-	-	-	-	150,306
Florida Mnt Strip Ratio	W:O	3.69	1.20	1.21	2.03	1.92	1.96	1.03													1.67
DeLamar Above COG	K Tonnes	-	-	2,250	4,529	133	9,661	15,194	10,440	8,487	2,423	1,420	2,441	2,003	2,033	2,037	2,108	2,011	53	-	67,221
	g Au/t	-	-	0.41	0.47	0.33	0.36	0.41	0.37	0.33	0.39	0.47	0.59	0.74	0.78	0.85	0.76	0.66	0.52	-	0.45
	K Ozs Au	-	-	30	69	1	111	201	123	89	30	21	46	47	51	56	51	43	1	-	972
	g Ag/t	-	-	20.46	41.68	18.63	13.20	18.74	27.62	37.27	48.01	42.09	59.42	64.64	68.76	67.24	61.19	55.04	46.00	-	33.08
	K Ozs Ag	-	-	1,480	6,068	80	4,101	9,155	9,271	10,171	3,741	1,921	4,663	4,162	4,494	4,403	4,146	3,558	79	-	71,491
DeLamar Waste	K Tonnes	-	-	4,107	2,287	113	9,328	18,403	23,182	24,363	30,427	31,430	15,546	8,782	5,503	3,246	1,717	731	11	-	179,174
DeLamar Total Mined	K Tonnes	-	-	6,357	6,815	246	18,989	33,597	33,621	32,850	32,850	32,850	17,987	10,785	7,535	5,282	3,825	2,742	64	-	246,395
DeLamar Strip Ratio	W:O			1.83	0.50	0.85	0.97	1.21	2.22	2.87	12.56	22.14	6.37	4.39	2.71	1.59	0.81	0.36	0.21		2.67
Total Above COG	K Tonnes	946	11,209	15,012	14,733	14,649	16,247	15,233	10,440	8,487	2,423	1,420	2,441	2,003	2,033	2,037	2,108	2,011	53	-	123,483
	g Au/t	0.35	0.53	0.42	0.44	0.44	0.39	0.41	0.37	0.33	0.39	0.47	0.59	0.74	0.78	0.85	0.76	0.66	0.52	-	0.45
	K Ozs Au	11	192	204	209	207	203	201	123	89	30	21	46	47	51	56	51	43	1	-	1,787
	g Ag/t	8.89	13.00	11.52	18.77	12.33	14.17	18.74	27.62	37.27	48.01	42.09	59.42	64.64	68.76	67.24	61.19	55.04	46.00	-	23.27
	K Ozs Ag	271	4,686	5,563	8,890	5,805	7,400	9,180	9,271	10,171	3,741	1,921	4,663	4,162	4,494	4,403	4,146	3,558	79	-	92,403
Total Waste	K Tonnes	3,491	13,498	19,566	22,996	28,025	22,261	18,443	23,182	24,363	30,427	31,430	15,546	8,782	5,503	3,246	1,717	731	11	-	273,217
Total Mined	K Tonnes	4,437	24,707	34,578	37,729	42,674	38,508	33,676	33,621	32,850	32,850	32,850	17,987	10,785	7,535	5,282	3,825	2,742	64	-	396,701
Total Strip Ratio	W:O	3.69	1.20	1.30	1.56	1.91	1.37	1.21	2.22	2.87	12.56	22.14	6.37	4.39	2.71	1.59	0.81	0.36	0.21		2.21
Material Processed																					
Total Leach	K Tonnes	610	10,287	12,548	12,635	11,072	12,600	12,600	10,914	6,325	997	296	450	259	176	170	200	103	0	-	92,241
	g Au/t	0.42	0.50	0.46	0.39	0.42	0.36	0.38	0.31	0.29	0.19	0.46	0.60	0.70	0.79	0.74	0.61	0.48	0.78	-	0.40
	K Ozs Au	8	165	186	158	148	147	155	110	59	6	4	9	6	4	4	4	2	0	-	1,175
	K Ozs Au Prod	-	119	137	118	104	105	102	81	45	13	2	6	3	3	3	2	1	0	-	844
	g Ag/t	10.55	13.43	12.06	13.64	11.77	13.65	18.51	21.57	33.16	44.62	24.02	57.84	77.92	77.57	78.07	74.82	71.64	143.50	-	17.30
	K Ozs Ag	207	4,443	4,865	5,541	4,188	5,528	7,498	7,568	6,744	1,431	229	837	648	439	426	480	236	0	-	51,310
	K Ozs Ag Prod	-	1,956	1,833	1,787	1,777	1,905	2,119	2,635	2,247	1,536	83	344	253	192	172	186	118	16	-	19,160
Total Mill	K Tonnes	-	-	-	1,982	2,160	2,160	2,160	2,166	2,160	2,160	2,160	2,166	2,160	2,160	2,160	2,166	2,160	1,162	-	31,243
	g Au/t	-	-	-	0.84	0.73	0.67	0.67	0.48	0.43	0.47	0.38	0.55	0.66	0.72	0.78	0.71	0.62	0.35	-	0.61
	K Ozs Au	-	-	-	53	51	46	47	33	30	33	27	38	46	50	54	50	43	13	-	612
	K Ozs Au Prod	-	-	-	39	41	36	20	14	13	14	13	16	19	20	22	20	18	7	-	311
	g Ag/t	-	-	-	43.80	24.30	24.21	19.85	36.94	44.41	38.21	30.18	54.37	52.11	59.28	58.07	53.41	48.25	14.01	-	40.91
	K Ozs Ag	-	-	-	2,792	1,687	1,681	1,379	2,572	3,084	2,653	2,096	3,786	3,619	4,117	4,033	3,719	3,351	524	-	41,093
	K Ozs Ag Prod	-	-	-	2,356	1,320	1,265	957	2,183	2,291	2,069	1,531	2,759	2,637	3,001	2,940	2,711	2,442	374	-	30,836
Total Project	K Tonnes	610	10,287	12,548	14,617	13,232	14,760	14,760	13,080	8,485	3,157	2,456	2,616	2,419	2,336	2,330	2,366	2,263	1,162	-	123,483
	g Au/t	0.42	0.50	0.46	0.45	0.47	0.41	0.43	0.34	0.32	0.38	0.39	0.56	0.66	0.72	0.78	0.70	0.62	0.35	-	0.45
	K Ozs Au	8	165	186	211	198	194	202	143	88	39	31	47	51	54	58	54	45	13	-	1,787
	K Ozs Au Prod	-	119	137	156	145	141	122	96	57	27	15	22	22	23	24	22	19	7	-	1,154
	g Ag/t	10.55	13.43	12.06	17.73	13.81	15.19	18.71	24.11	36.03	40.23	29.44	54.96	54.87	60.66	59.53	55.22	49.31	14.01	-	23.27
	K Ozs Ag	207	4,443	4,865	8,333	5,876	7,209	8,877	10,141	9,828	4,084	2,325	4,623	4,267	4,556	4,459	4,200	3,587	524	-	92,403
	K Ozs Ag Prod	-	1,956	1,833	4,142	3,097	3,170	3,077	4,818	4,538	3,605	1,614	3,103	2,890	3,193	3,112	2,897	2,560	390	-	49,996

Figure 22-2 Gold Equivalent Production Profile by Process Method

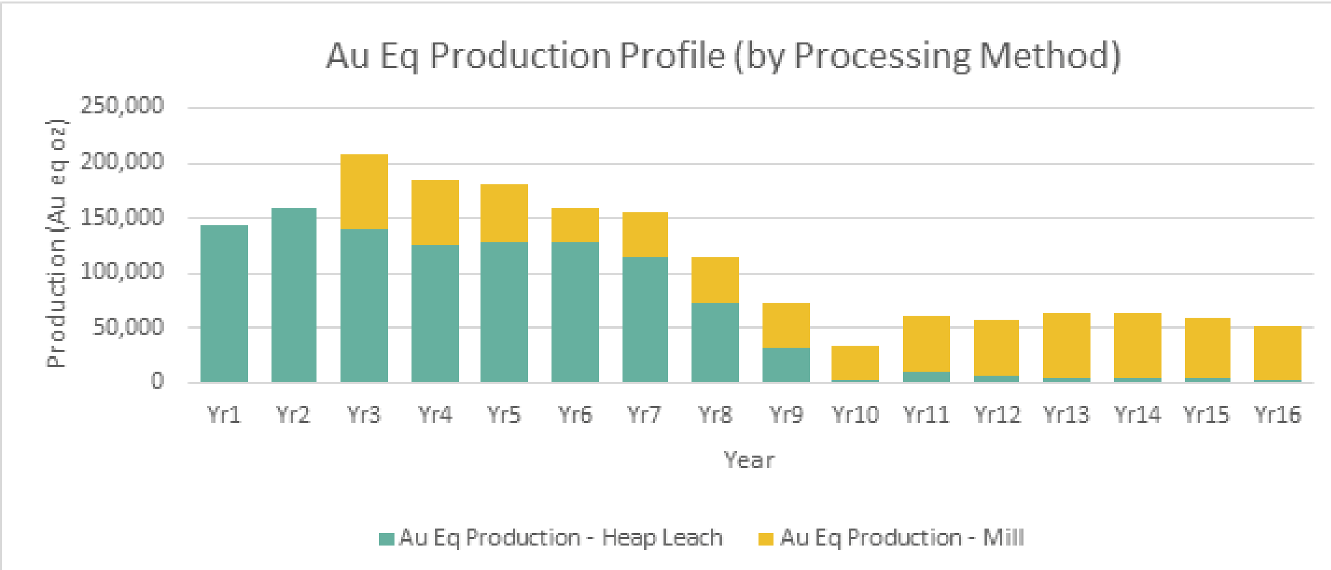
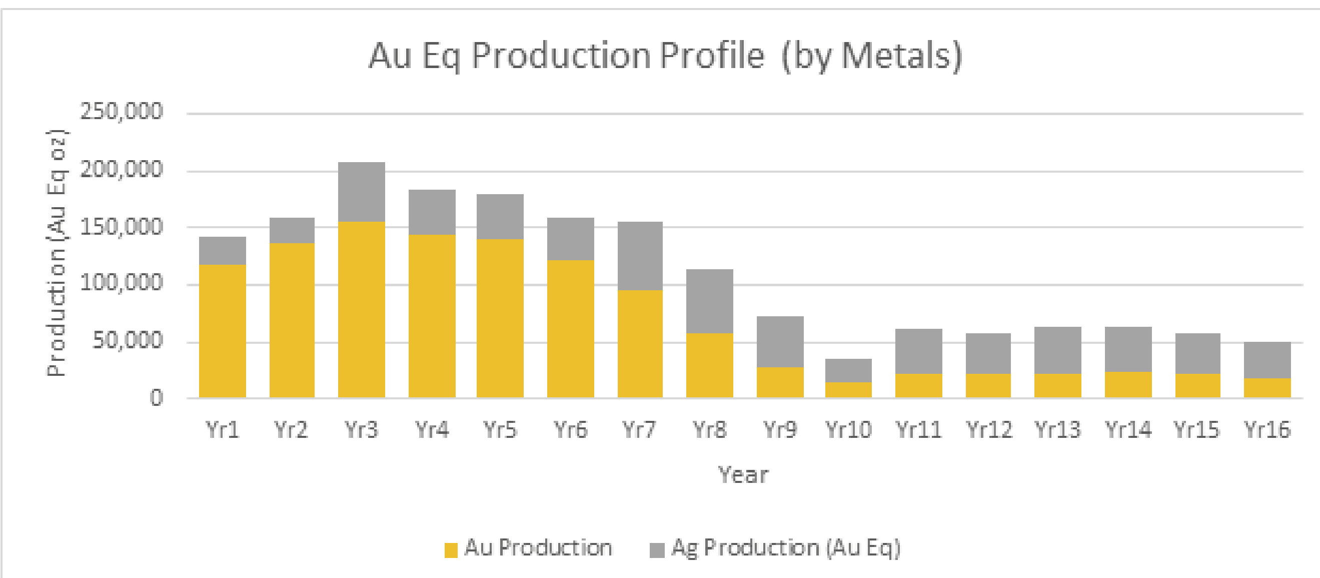


Figure 22-3 Gold Equivalent Profile by Process Metals



22.2 Pre-Tax Cash Flow

The pre-tax cash-flow model is shown in Table 22.2. This is based on the mining physicals shown in Table 22.1 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 $1,700 and $21.50 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 Ag of silver produced. Royalties have been applied as NSR royalties described in Section 4.3.

Table 22-2 Pre-Tax Cash Flow

Revenues	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Payable Metal - Au	K Ozs Au	-	118	136	156	145	141	121	95	57	27	15	22	22	23	24	22	19	7	-	1,149
Gold Revenue	K USD	$ -	$ 200,556	$ 231,218	$ 264,601	$ 245,875	$238,857	$205,662	$ 161,985	$ 97,012	$ 45,524	$ 24,715	$ 37,718	$ 37,060	$ 38,881	$ 41,206	$ 37,583	$ 31,809	$ 12,459	$ -	$1,952,721
Transp. & Refining - Au	K USD	$ -	$ 590	$ 680	$ 778	$ 723	$ 703	$ 605	$ 476	$ 285	$ 134	$ 73	$ 111	$ 109	$ 114	$ 121	$ 111	$ 94	$ 37	$ -	$ 5,743
Payable Metal - Ag	K Ozs Ag	-	1,946	1,824	4,122	3,082	3,154	3,061	4,794	4,515	3,587	1,606	3,088	2,875	3,177	3,097	2,883	2,548	388	-	49,746
Silver Revenue	K USD	$ -	$ 41,837	$ 39,214	$ 88,615	$ 66,254	$ 67,815	$ 65,817	$ 103,075	$ 97,077	$ 77,126	$ 34,525	$ 66,381	$ 61,815	$ 68,316	$ 66,577	$ 61,982	$ 54,772	$ 8,335	$ -	$1,069,532
Transp. & Refining - Ag	K USD	$ -	$ 973	$ 912	$ 2,061	$ 1,541	$ 1,577	$ 1,531	$ 2,397	$ 2,258	$ 1,794	$ 803	$ 1,544	$ 1,438	$ 1,589	$ 1,548	$ 1,441	$ 1,274	$ 194	$ -	$ 24,873
Payable Gold Equivalent	K Ozs AuEq	-	143	159	208	184	180	160	156	114	72	35	61	58	63	63	59	51	12	-	1,778
Revenue Before Royalties	K USD	$ -	$ 240,830	$ 268,840	$ 350,377	$ 309,865	$304,393	$269,344	$ 262,186	$ 191,545	$ 120,723	$ 58,364	$ 102,444	$ 97,328	$ 105,494	$ 106,113	$ 98,013	$ 85,213	$ 20,564	$ -	$2,991,636
Royalties	K USD	$ -	$ 532	$ 2,976	$ 3,879	$ 1,334	$ 2,897	$ 2,665	$ 2,528	$ 2,305	$ 833	$ 551	$ 973	$ 944	$ 1,018	$ 1,029	$ 956	$ 813	$ 137	$ -	$ 26,369
Net Revenue	K USD	$ -	$ 240,298	$ 265,864	$ 346,498	$ 308,532	$301,496	$266,679	$ 259,658	$ 189,241	$ 119,890	$ 57,813	$ 101,471	$ 96,384	$ 104,475	$ 105,084	$ 97,057	$ 84,401	$ 20,427	$ -	$2,965,267
Operating Costs																					
Mining Costs	K USD	$ -	$ 49,572	$ 63,439	$ 72,691	$ 70,702	$ 71,414	$ 62,265	$ 57,939	$ 53,120	$ 54,874	$ 55,338	$ 37,275	$ 26,447	$ 19,881	$ 16,768	$ 15,344	$ 14,044	$ 2,292	$ -	$ 743,405
Processing Costs																					
Leaching	K USD	$ -	$ 37,933	$ 45,272	$ 45,495	$ 42,034	$ 46,147	$ 45,423	$ 40,047	$ 25,940	$ 4,032	$ 1,673	$ 2,242	$ 1,553	$ 1,219	$ 1,263	$ 1,424	$ 1,042	$ 64	$ -	$ 342,804
Milling	K USD	$ -	$ -	$ -	$ 23,263	$ 24,397	$ 24,592	$ 27,512	$ 26,664	$ 27,664	$ 27,341	$ 27,187	$ 28,232	$ 28,100	$ 28,280	$ 28,265	$ 28,391	$ 28,276	$ 14,435	$ -	$ 392,598
Total Processing Cost	K USD	$ -	$ 37,933	$ 45,272	$ 68,758	$ 66,431	$ 70,739	$ 72,935	$ 66,711	$ 53,604	$ 31,372	$ 28,860	$ 30,474	$ 29,653	$ 29,498	$ 29,527	$ 29,816	$ 29,318	$ 14,499	$ -	$ 735,402
Other Costs																					
G&A	K USD	$ -	$ 7,328	$ 6,948	$ 8,548	$ 8,279	$ 8,010	$ 7,740	$ 7,471	$ 7,202	$ 6,322	$ 6,053	$ 5,784	$ 4,923	$ 4,732	$ 4,732	$ 4,732	$ 4,732	$ 2,125	$ -	$ 105,659
Reclamation - Florida Mnt	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation - DeLamar	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Reclamation	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Operating Cost	K USD	$ -	$ 94,834	$ 115,659	$ 149,997	$ 145,411	$150,163	$142,940	$ 132,121	$ 113,926	$ 92,569	$ 90,251	$ 73,532	$ 61,023	$ 54,111	$ 51,027	$ 49,892	$ 48,094	$ 18,916	$ -	$1,584,466
Net Operating Cash Flow	K USD	$ -	$ 145,464	$ 150,205	$ 196,500	$ 163,121	$151,333	$123,738	$ 127,537	$ 75,315	$ 27,321	$ (32,438)	$ 27,939	$ 35,362	$ 50,364	$ 54,057	$ 47,166	$ 36,307	$ 1,510	$ -	$1,380,801
Cumulative Op Cash Flow	K USD		$ 145,464	$ 295,669	$ 492,169	$ 655,290	$806,623	$930,362	$1,057,899	$1,133,214	$1,160,535	$1,128,097	$1,156,036	$1,191,397	$1,241,762	$1,295,819	$1,342,984	$1,379,291	$1,380,801		
Capital Costs																					
Pre-Stripping	K USD	$ 12,712	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 12,712
Pre-Production Op Costs	K USD	$ 7,001	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 7,001
Mining Capital	K USD	$ 30,778	$ 9,159	$ 8,911	$ 8,934	$ 10,963	$ 18,969	$ 10,225	$ 10,536	$ 11,073	$ -	$ -	$ -	$ -	$ 0	$ -	$ -	$ -	$ -	$ -	$ 119,548
Process Capital	K USD	$ 211,330	$ 41,514	$ 128,119	$ 11,084	$ 49	$ 15,187	$ 44	$ -	$ -	$ -	$ 14,678	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 422,005
Infrastructure / Owner's Capital	K USD	$ 20,109	$ -	$ -	$ -	$ -	$ 974	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 21,083
Reclamation	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 30,835	$ -	$ 30,835
Sub-Total	K USD	$ 281,930	$ 50,673	$ 137,029	$ 20,018	$ 11,012	$ 35,130	$ 10,269	$ 10,536	$ 11,073	$ -	$ 14,678	$ -	$ -	$ 0	$ -	$ -	$ -	$ 30,835	$ -	$ 613,184
Working Capital	K USD	$ -	$ 19,518	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (19,518)	$ -	$ -
Bonding Capital	K USD	$ 6,167	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (6,167)	$ -	$ -
Salvage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (23,729)	$ -	$ (23,729)
Total Capital	K USD	$ 288,097	$ 70,192	$ 137,029	$ 20,018	$ 11,012	$ 35,130	$ 10,269	$ 10,536	$ 11,073	$ -	$ 14,678	$ -	$ -	$ 0	$ -	$ -	$ -	$ (18,579)	$ -	$ 589,455
Pre-Tax Cash Flow	K USD	$ (288,097)	$ 75,272	$ 13,176	$ 176,482	$ 152,109	$116,203	$113,469	$ 117,001	$ 64,242	$ 27,321	$ (47,116)	$ 27,939	$ 35,362	$ 50,364	$ 54,057	$ 47,166	$ 36,307	$ 20,090	$ -	$ 791,346
Cumulative Pre-Tax Cash Flow	K USD	$ (288,097)	$(212,825)	$(199,649)	$ (23,167)	$ 128,942	$245,146	$358,615	$ 475,616	$ 539,858	$ 567,179	$ 520,063	$ 548,002	$ 583,363	$ 633,727	$ 687,784	$ 734,950	$ 771,256	$ 791,346		
Pre-Tax Payback	Years		1.00	1.00	1.00	0.15	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.15

Pre-tax NPV (5%)	K USD	$478,377
Pre-tax NPV (8%)	K USD	$355,604
Pre-tax NPV (10%)	K USD	$291,266
Pre-tax IRR	%	30%

The total net revenue prior to application of costs is $2.97 billion. Subtracting the total operating costs of $1.58 billion and estimated capital costs of $589.5 million yields a pre-tax LOM cash flow of $791.3 million (apparent discrepancies are due to rounding). The pre-tax LOM NPV at a 5.0% discount is estimated at $478.4 million, for a pre-tax IRR of 30%.

22.3 Tax Considerations & After-Tax Cash Flow

RESPEC 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 pre-tax LOM cash flow are shown in Table 22.3.

Existing credits, NOL, deductions, depreciation and depletion reduce the taxable income to $339.2 million. Estimated federal taxes (21%), Idaho State Tax (6.5%), and Idaho Mining Tax (1%) total $102.1 million. After deductions and credits, the resulting effective tax rate is 13%.

Tax calculations and the after-tax cash flow are shown in Table 22.3. The after-tax cash flow is $689.3 million or NPV(5%) of $407.8 million with a 27% IRR and a 3.3-year return on investment.

Table 22-3 Depreciation, Depletion, Taxes, and After-Tax Cash Flow

	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Yr_11	Yr_12	Yr_13	Yr_14	Yr_15	Yr_16	Yr_17	Yr_18	Total
Forecasted Cash Flow before CapEx and Tax	**K USD**	**$ -**	**$ 145,464**	**$ 150,205**	**$ 196,500**	**$ 163,121**	**$ 151,333**	**$ 123,738**	**$ 127,537**	**$ 75,315**	**$ 27,321**	**$ (32,438)**	**$ 27,939**	**$ 35,362**	**$ 50,364**	**$ 54,057**	**$ 47,166**	**$ 36,307**	**$ 1,510**	**$ -**	**$ 1,380,801**
Less: Depreciation	K USD	$ -	$ (169,946)	$ (56,981)	$ (61,360)	$ (51,127)	$ (49,750)	$ (51,619)	$ (50,703)	$ (36,715)	$ (14,721)	$ (6,709)	$ (6,792)	$ (4,363)	$ (2,221)	$ (2,049)	$ (1,883)	$ (1,798)	$ (900)	$ -	$ (569,637)
Less: Amortization of Permitting	K USD	$ -	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ (239)	$ -	$ -	$ -	$ (3,587)
Less: Amortization of Exploration	K USD	$ (2,425)	$ (2,425)	$ (2,407)	$ (1,745)	$ (923)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (9,926)
Less: Amortization of Development	K USD	$ -	$ (1,020)	$ (1,137)	$ (1,486)	$ (1,313)	$ (1,290)	$ (1,142)	$ (1,115)	$ (816)	$ (516)	$ (249)	$ (438)	$ (416)	$ (451)	$ (453)	$ (419)	$ (364)	$ (87)	$ -	$ (12,712)
Less: Reclamation expenditures	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (30,835)	$ -	$ (30,835)
Less: state income and mine license tax	K USD	$ -	$ (3,588)	$ (2,304)	$ (4,386)	$ (3,450)	$ (3,079)	$ (1,980)	$ (2,158)	$ (1,071)	$ (444)	$ -	$ (235)	$ (371)	$ (1,195)	$ (2,685)	$ (2,272)	$ (1,626)	$ -	$ -	$ (30,844)
Plus: Equipment Salvage Gain	K USD											$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 23,729	$ -	$ 23,729
Net Income before Depletion	**K USD**	**$ (2,425)**	**$ (31,754)**	**$ 87,137**	**$ 127,284**	**$ 106,068**	**$ 96,974**	**$ 68,758**	**$ 73,322**	**$ 36,473**	**$ 11,401**	**$ (39,635)**	**$ 20,235**	**$ 29,974**	**$ 46,258**	**$ 48,631**	**$ 42,353**	**$ 32,518**	**$ (6,583)**	**$ -**	**$ 746,989**
Less: Depletion Allowed	K USD	$ -	$ (986)	$ (39,271)	$ (51,180)	$ (45,577)	$ (44,534)	$ (34,379)	$ (36,661)	$ (18,237)	$ (5,700)	$ -	$ (10,117)	$ (14,235)	$ (15,430)	$ (15,520)	$ (14,335)	$ (12,465)	$ -	$ -	$ (358,629)
Less: NOL / Foreign Derived Intangible Income Deduction	K USD	$ 2,425	$ 32,740	$ (38,293)	$ (20,437)	$ (4,987)	$ (3,998)	$ (1,001)	$ (2,679)	$ -	$ -	$ 39,635	$ (8,094)	$ (12,715)	$ (21,171)	$ (6,976)	$ (5,945)	$ (4,253)	$ 6,583	$ -	$ (49,164)
Net Taxable Income (Federal)	**K USD**	**$ -**	**$ -**	**$ 9,573**	**$ 55,667**	**$ 55,504**	**$ 48,443**	**$ 33,378**	**$ 33,982**	**$ 18,237**	**$ 5,700**	**$ -**	**$ 2,023**	**$ 3,024**	**$ 9,657**	**$ 26,135**	**$ 22,074**	**$ 15,800**	**$ -**	**$ -**	**$ 339,197**
Federal Taxes (21%)	K USD	$ -	$ -	$ 2,010	$ 11,690	$ 11,656	$ 10,173	$ 7,009	$ 7,136	$ 3,830	$ 1,197	$ -	$ 425	$ 635	$ 2,028	$ 5,488	$ 4,635	$ 3,318	$ -	$ -	$ 71,231
Idaho State Tax (6.5%)	K USD	$ -	$ 2,917	$ 1,996	$ 3,801	$ 2,990	$ 2,669	$ 1,716	$ 1,870	$ 928	$ 385	$ -	$ 133	$ 209	$ 875	$ 2,327	$ 1,969	$ 1,409	$ -	$ -	$ 26,196
Idaho Mining Tax (1%)	K USD	$ -	$ 671	$ 307	$ 585	$ 460	$ 411	$ 264	$ 288	$ 143	$ 59	$ -	$ 102	$ 161	$ 320	$ 358	$ 303	$ 217	$ -	$ -	$ 4,648
Total Tax	**K USD**	**$ -**	**$ 3,588**	**$ 4,314**	**$ 16,076**	**$ 15,106**	**$ 13,252**	**$ 8,989**	**$ 9,294**	**$ 4,901**	**$ 1,641**	**$ -**	**$ 660**	**$ 1,006**	**$ 3,223**	**$ 8,173**	**$ 6,907**	**$ 4,944**	**$ -**	**$ -**	**$ 102,076**
Pre-tax Cash Flow	K USD	$ (288,097)	$ 75,272	$ 13,176	$ 176,482	$ 152,109	$ 116,203	$ 113,469	$ 117,001	$ 64,242	$ 27,321	$ (47,116)	$ 27,939	$ 35,362	$ 50,364	$ 54,057	$ 47,166	$ 36,307	$ 20,090	$ -	$ 791,346
Total Tax	K USD	$ -	$ 3,588	$ 4,314	$ 16,076	$ 15,106	$ 13,252	$ 8,989	$ 9,294	$ 4,901	$ 1,641	$ -	$ 660	$ 1,006	$ 3,223	$ 8,173	$ 6,907	$ 4,944	$ -	$ -	$ 102,076
Net After Tax Cash Flow	K USD	$ (288,097)	$ 71,684	$ 8,862	$ 160,406	$ 137,003	$ 102,951	$ 104,480	$ 107,707	$ 59,341	$ 25,680	$ (47,116)	$ 27,279	$ 34,356	$ 47,141	$ 45,884	$ 40,258	$ 31,363	$ 20,090	$ -	$ 689,270
Cumulative After Tax Cash Flow	K USD	$ (288,097)	$ (216,413)	$ (207,550)	$ (47,145)	$ 89,858	$ 192,809	$ 297,289	$ 404,996	$ 464,337	$ 490,017	$ 442,901	$ 470,179	$ 504,535	$ 551,676	$ 597,560	$ 637,818	$ 669,181	$ 689,270		
After Tax Payback Calculation	Years		1.00	1.00	1.00	0.34	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.34

After-tax NPV (5%)	K USD	$407,817
After-tax NPV (8%)	K USD	$297,519
After-tax NPV (10%)	K USD	$239,771
After-tax IRR	%	27%
After-Tax Payback Period	Years	3.34

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 22.4. Sensitivity to operating costs, and capital costs were evaluated and are shown in Table 22.5 and Table 22.6, respectively, showing the after-tax values. The after-tax sensitivity to revenues, capital, and operating costs is shown in Figure 22.4.

Table 22-4 Project Sensitivity to Metal Prices

$/oz Au	$/oz Ag	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
$ 1,500	$ 18.97	$198,811	$123,406	$84,281	16%	4.30
$ 1,550	$ 19.60	$251,296	$167,213	$123,450	19%	3.94
$ 1,600	$ 20.24	$304,035	$211,159	$162,701	22%	3.72
$ 1,650	$ 20.87	$355,830	$254,247	$201,148	24%	3.52
$ 1,700	$ 21.50	$407,817	$297,519	$239,771	27%	3.34
$ 1,750	$ 22.13	$459,528	$340,561	$278,192	29%	3.19
$ 1,800	$ 22.76	$510,589	$383,015	$316,060	32%	3.05
$ 1,850	$ 23.40	$561,343	$425,183	$353,653	34%	2.93
$ 1,900	$ 24.03	$611,998	$467,275	$391,183	36%	2.83
$ 1,950	$ 24.66	$662,697	$509,428	$428,785	39%	2.73
$ 2,000	$ 25.29	$713,650	$551,851	$466,659	41%	2.64

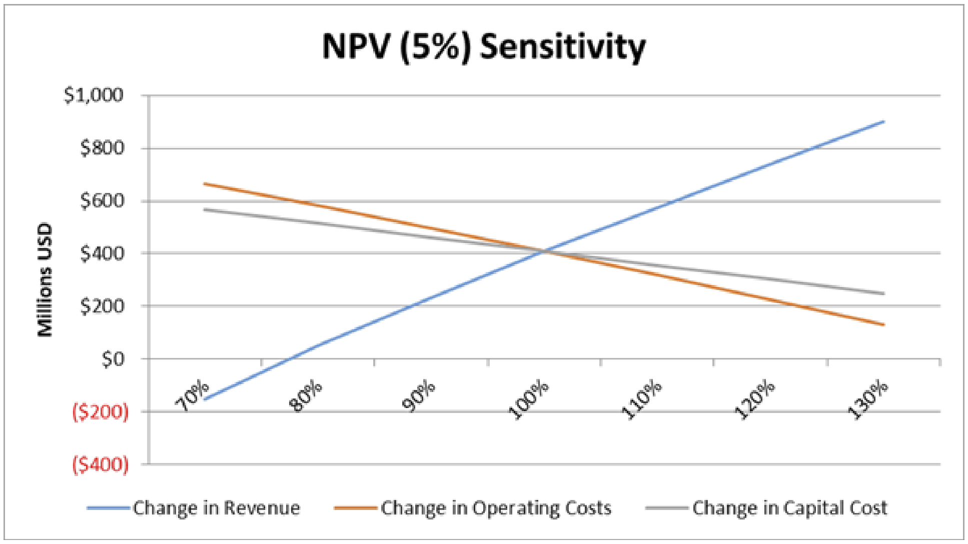
Table 22-5 Operating Cost Sensitivity (After Tax)

% Change	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
70%	$665,895	$508,579	$426,063	37%	2.82
80%	$581,380	$439,464	$365,055	34%	2.96
90%	$495,871	$369,597	$303,426	31%	3.13
100%	$407,817	$297,519	$239,771	27%	3.34
110%	$318,066	$224,042	$174,883	23%	3.60
120%	$226,328	$148,715	$108,254	18%	3.92
130%	$130,877	$70,774	$39,513	13%	4.53

Table 22-6 Capital Cost Sensitivity (After Tax)

% Change	NPV (5%)	NPV (8%)	NPV (10%)	IRR	Payback
70%	$566,864	$447,197	$383,738	48%	2.35
80%	$513,848	$397,304	$335,749	39%	2.66
90%	$460,833	$347,411	$287,760	32%	2.97
100%	$407,817	$297,519	$239,771	27%	3.34
110%	$354,802	$247,626	$191,781	22%	3.73
120%	$301,786	$197,733	$143,792	18%	4.16
130%	$248,771	$147,840	$95,803	15%	4.74

Figure 22-4 After-Tax Sensitivity



23.0 ADJACENT PROPERTIES

The authors have no information to report from adjacent properties.

24.0 OTHER RELEVANT DATA AND INFORMATION

The authors have no other relevant data and information to report.

25.0 INTERPRETATION AND CONCLUSIONS

Work completed by Integra at the DeLamar project since its acquisition in 2017 has led to the estimation of mineral resources, which have been updated several times, a mineral reserve estimate and several economic studies with the results of the most recent PFS being reproduced in this technical report. The current resources described herein include updated in-situ resources and first-time estimates of stockpile resources. The stockpiles are comprised of materials that were mined during historical open-pit operations but not processed, and they include historical waste-rock storage facilities stacked near the historical open-pits, as well as backfill into some of the pits, at both the DeLamar and Florida Mountain areas. The effective date of the mineral resources included in this report is August 25, 2023.

As noted, this technical report also includes the results of a PFS and mineral reserve statement on the DeLamar project included in the NI 43-101 technical report titled "Technical Report and Preliminary Feasibility Study for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated March 22, 2022 with an effective date of January 24, 2022.

Mr. Dyer, P.E., the responsible qualified person for the mineral reserve estimate in the aforementioned technical report and included in this technical report, reviewed the updated mineral resource model and determined that the updated mineral resource model does not materially change the mineral reserve statement included in the aforementioned technical report. Accordingly, the results of the PFS and the mineral reserve statement have been reproduced in this technical report and remain unaffected by the updated mineral resource. The PFS and mineral reserve statement have an effective date of January 24, 2022.

The PFS total LOM gold production is estimated to be 1,154,000 ounces, with LOM average recovery of 72% for the heap leach and 51% for the mill. Silver production is estimated to be 50.0 million ounces, with an average LOM recovery of 37% for the heap leach and 75% for the mill. The DeLamar area pits have an overall strip ratio of 2.67 tonnes of waste per tonne processed. The Florida Mountain pit has a stripping ratio of 1.67 tonnes of waste per tonne processed.

Economic highlights of the PFS include (i) after-tax NPV (5%) of $407.8 million with a 27% after-tax IRR using prices of $1,700 and $21.50 per ounce gold and silver, respectively; (ii) after-tax payback period of 3.34 years; and (iii) year 1 to 8 average production of 163,000 oz AuEq (121,000 oz Au and 3,312,000oz Ag), with year 1 to 16 average production of 110,000 oz AuEq (71,000 oz Au and 3,085,000 oz Ag). The total cash cost is estimated to be $923 per oz AuEq, with site level all-in sustaining costs estimated to be $955 per oz AuEq.

A gold price of $1,650 per ounce gold price was used to determine the ultimate pit limits for design, while a gold price of $1,700 per ounce was used for the PFS economic evaluation. The statement of Proven and Probable reserves is supported by the positive economic evaluation.

The DeLamar project gold and silver deposits are characterized as volcanic-hosted, low-sulfidation epithermal mineralization. At the DeLamar area, presently defined near-surface gold and silver resources extend continuously for approximately three kilometers (1.89 miles) of strike length, a maximum northeast-southwest width of 1.1 kilometers (0.69 miles), and an elevation range of 450 meters (1,476 feet). The Milestone area of the DeLamar mineralization adds an additional 670 meters (2,198 feet) of strike to the resources. The near-surface resources at Florida Mountain have a northerly strike extent of about 1.3 kilometers (0.81 miles), an east-west width of up to 650 meters (2,133 feet), and an elevation range of 400 meters (1,312 feet).

Metallurgical testing has shown that oxide and mixed mineralization types 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 with agglomeration required for a significant portion of the DeLamar area heap-leachable mineralization. Non-oxide mineralization from the Florida Mountain and DeLamar deposits is amenable to grinding followed by flotation, flotation concentrate regrind, and agitated cyanide leaching of the reground concentrate for recovery of gold and silver. Gold and silver recoveries from Florida Mountain non-oxide average 83% and 72%, respectively. Gold recoveries from the DeLamar non-oxide material are variable and generally low because some of the contained gold is locked in sulfide minerals. The average silver recovery from the DeLamar non-oxide material is 75%, which leads to silver making a significant contribution to the project economics. Preliminary testing has indicated good potential for heap leach cyanidation processing of the historical dumps and stockpiles. Preliminary recovery estimates are similar to those for the in-situ oxide and mixed material types.

Potential open-pit gold and silver resources at the DeLamar project, inclusive of the mineral reserves, were constrained to lie within optimized pits and were tabulated using cutoff grades of 0.17 g AuEq/t for oxide and mixed materials at both the DeLamar and Florida Mountain areas, 0.10 g AuEq/t for all stockpile materials, 0.3 g AuEq/t for non-oxide mineralization at the DeLamar area, and 0.2 g AuEq/t for non-oxide materials at Florida Mountain. Project-wide Measured and Indicated resources total 247,836,000 tonnes averaging 0.37 g Au/t (2,935,000 ounces of gold) and 18.1 g Ag/t (142,748,000 ounces of silver). Inferred resources total 43,101,000 tonnes at an average grade of 0.31 g Au/t (428,000 ounces of gold) and 10.8 g Ag/t (15,002,000 ounces of silver). Total Proven and Probable reserves for the DeLamar project from all pit phases remain unchanged from those reported in 2022 at 123,483,000 tonnes with average grades of 0.45 g Au/t and 23.27 g Ag/t, for 1,787,000 ounces of gold and 92,403,000 ounces of silver.

The total project resources include first-time estimates of the stockpile materials. Measured and Indicated stockpile resources total 42,555,000 tonnes averaging 0.22 g Au/t (296,000 ounces of gold) and 11.8 g Ag/t (16,149,000 ounces of silver). Inferred stockpile resources are comprised of 4,877,000 tonnes that average 0.17 g Au/t (26,000 ounces of gold) and 9.8 g Ag/t (1,535,000 ounces of silver). Approximately 70% of the full tonnage of the historical stockpile materials has been drilled to date. Further drilling will be required to define the full extents of the stockpile resources.

The authors have reviewed the data from the DeLamar project, which includes the DeLamar and Florida Mountain areas, and have undertaken verification of the data that is material to this report. Based on the work completed or supervised by the authors, the authors have determined that the project data are of sufficient quality for the purposes used in this report. 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 and mineral reserves other than those discussed herein.

25.1 DeLamar Project Opportunities

Additional project opportunities presented in the PFS and reproduced herein are still viable, and Integra is actively working to implement them in future technical studies.

25.1.1 Heap Leach Stage 1

There is the potential to lower project capital costs by foregoing mill processing and instead operate a heap-leach only project. In this scenario, a high percentage of the current heap-leach reserves would be processed at the 35,000 tonne per day rate envisioned in the PFS. LOM capital expenditures would decrease significantly as expansion capital, such as non-oxide plant and tailing facilities, would not be required. A decision to construct and initiate mill processing (Stage 2) could be exercised at any time, providing the flexibility to respond to changing market conditions and thereby reduce project risk.

A heap-leach only approach could reduce risk and provide greater flexibility to respond to the prevailing economic environment in connection with a decision to pursue a milling scenario later.

25.1.2 Exploration for Underground Development

From 1891 through 1998, total historical production of gold and silver from the DeLamar 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 at the DeLamar area from the original 'De Lamar' underground mine and the later open-pit operations. At Florida Mountain, nearly 260,000 ounces of gold and 18 million ounces of silver were produced from historical underground mining and late-1990s open-pit mining.

There is potential for the discovery of unmined high-grade vein-type mineralization at the DeLamar project that may lie below the known, near-surface deposits. This is especially true at Florida Mountain, where 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 (5,905 feet) and vertical extents up to 450 meters (1,476 feet). 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 that may exist. Just as lower-grade mineralization overlies the historical stopes of the Trade Dollar-Black Jack vein system, the current resources include similar low-grade mineralization below which vein structures analogous to the Trade Dollar-Black Jack have only been partially explored. Integra now has over 100 drill intercepts grading 4 g AuEq/t or greater over widths of 1.52 meters (5.0 feet) or greater, which speaks to potential higher-grade mineralization at depth at Florida Mountain.

At the DeLamar area, historical underground and open-pit mining exploited high-grade veins in the Sommercamp and North DeLamar zones, which include less than 500 meters of the total three kilometers of strike length of continuous DeLamar area near-surface mineralization. The Milestone area adds another 0.6 kilometers of near-surface mineralization that lacks testing for deeper high-grade zones. In both of these areas, Integra has reported high-grade gold-silver intercepts that support the potential for discovery of high-grade vein-type mineralization.

25.1.3 Other Opportunities

Opportunities to improve process recoveries and/or decrease process costs through continued metallurgical testing include:

- Evaluation of ROM leaching for lower-grade oxide materials;
- Further optimization of the planned heap-leach and mill processes may improve recoveries and/or decrease reagent consumptions;
- Continued evaluation of higher-grade oxide and mixed material types (particularly for silver) for processing by grind-leach and flotation with concentrate regrind and leach, to determine if any of these materials are better processed by milling; and
- Ongoing optimization of the geo-metallurgical model for further optimization of ore routing to improve recoveries.

The opportunity to add value to the project through the processing of the DeLamar non-oxide materials will include evaluation of the following:

- Further studies on oxidative pretreatment options such as Albion processing should be advanced with the goal of improving metals recoveries and project economics. Scoping-level Albion test results have yielded gold and silver recoveries of >80% for Sullivan Gulch and >70% for Glen Silver;
- Evaluation and optimization of flotation concentrate processing;
- Further testwork to investigate high-density or paste deposition of the flotation tailing, which could lessen the footprint and risk of the associated TSF. Opportunities to generate power from the tailing being pumped to the TSF will also be investigated.

As with all gold and silver projects, an increase in metal prices will reflect an upside in total revenues for the project.

25.1.4 Environment, Social, and Governance ("ESG")

As the mining industry evolves to supply materials safely and responsibly to a changing modern world, strong emphasis has been placed on environment, social, and governance ("ESG") factors. Integra has engaged its core value of *Innovation* to guide the process of evaluating the many aspects of the DeLamar project's plan. Integra will continue to evaluate technologies and solutions that highlight how the project can benefit economically from taking steps towards sustainability, such as potential vehicle electrification and handling of ancillary waste.

25.1.4.1 Equipment and Vehicle Electrification

The total amount of diesel used by the haul trucks for the DeLamar site as envisioned by the PFS is estimated to be 126.1 megaliters (33.4 million gallons) per year, presenting a significant opportunity for emissions reductions. Total emissions from diesel use over the life of operations is estimated at 1.1 million tonnes. In addition to reducing emissions, alternatives to diesel potentially offer significant savings in capital and operating costs.

Rapid technology development around electric motors and drives in mining equipment provides the industry with opportunities to transition conventional diesel equipment to more energy and cost-efficient alternatives. Replacing diesel motors with electric motors will reduce the mine's total and variability of energy spend. Multiple models of high-capacity battery electric vehicle ("BEV") haul trucks are currently being tested in the mining industry. While BEVs have been deployed in underground mining for years, several manufacturers such as Komatsu, Hitachi, Belaz, etc., have been developing large-haul BEVs for open-pit applications and are aiming for full commercialization within the next five years.

As an alternative to diesel, BEVs offer the most comparable, low-emissions option, along with hydrogen-powered vehicles. BEVs also improve noise and air quality surrounding the operational site. According to a February 2022 presentation by Sandvik at the Electric Mine Conference in Stockholm Sweden, BEVs not only require less in operating and maintenance costs but can be up to twice as fast at steeper grades compared to their diesel counterparts.

Hydrogen-powered vehicles offer the same reduction in fuel emissions as BEVs. While a smaller number of manufacturers are developing hydrogen-powered mining vehicles, two models are being tested in 2022. Estimated upfront costs are comparable to a typical diesel vehicle. Currently, the main barrier to hydrogen-powered vehicles is the development stage of the required infrastructure. As development continues, easier access to the necessary supply chains or onsite hydrogen production may become available.

25.1.4.2 Ancillary Waste (Office, Truck Shop, Etc.)

The DeLamar project can be conservatively expected to generate roughly 176.6 tonnes of non-productive waste, or waste generated by staff on-site, per year. Only 29.3 tonnes per year (16.5%) of this waste is estimated to be landfill waste, presenting a significant opportunity for diversion on the site. Approximately 48.8 tonnes (27.6%) annually are expected to be organic waste, and the remaining inorganic recyclable waste is estimated to be 98.4 tonnes per year (55.7%). The proportion of organic waste presents an opportunity to avoid costs associated with taking waste off-site while reducing associated emissions. The production of compost material can be used on tailing piles as a nutrient-dense soil to support revegetation.

25.1.5 Geotechnical Optimization

Additional geotechnical studies will be completed in the next year which could lead to steeper pit slopes and reduced stripping requirements.

25.2 DeLamar Project Risks

The risks described in this section may potentially have an impact on the economic viability or continued viability of the DeLamar Project.

25.2.1 Operating Risks

Risks related to the metallurgy/mineral processing include:

- Expected heap-leach gold and silver recoveries from the DeLamar mixed mineralization and expected mill gold recoveries from the DeLamar non-oxide mineralization are variable. This reduces the certainty of future revenue streams. Further variability testing is planned to improve recovery modelling and to optimize ore routing;
- Variability testing may reveal a need to reclassify some materials to different oxidation categories, which can adversely affect expected recoveries;
- Though some agglomeration is planned for DeLamar, problems related to high-clay content in certain DeLamar material types may be severe enough to adversely affect projected heap-leach and mill recoveries. Further characterization testing of the elevated-clay material types is planned so that that these potential challenges can be adequately dealt with by well-planned ore control and metallurgical processing;
- Further geotechnical testing will be needed, especially for the HLP, the processing facilities, and the pit high walls. There is a risk that this testing may identify less favorable geotechnical parameters than have been used in the PFS;
- The site hydrogeology is not yet well understood. Further work is required to better understand the site-wide water balance and operational needs; and
- Geochemical analysis of both ore and waste is needed to better define the acid-generating potential of the various material types to ensure proper lime dosing in the heap and potential routing of higher sulfide mixed mineralization to the mill.

The mine site is located at 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 if unusually heavy snowfall occurs.

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.

25.2.2 Permitting Risks and Risk Management Strategy

This section summarizes certain environmental issues and risks, as well as strategies 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 Integra, 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 ("BMPs") into their exploration plans. These programs would be carried over by Integra to any full-scale mining operation. In addition, Integra's permitting 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 based on input from the second step. This could involve: 1) operational monitoring; 2) reclamation planning; 3) employment and business opportunities; 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.

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 water treatment to meet these criteria. The water treatment facilities contemplated in the PFS have been proven at other mining operations located in very sensitive environments.

25.2.3 Climate Change Risks

Climate change driven risks to the DeLamar project are broadly common to other mining operations in the United States intermountain region. Risks include direct operational impacts associated with changes to site hydrology and water supply, temperature/meteoric conditions, and more general climate-driven risks that include temperature-related decreases in workforce productivity/safety, propensity for wildfires, regional infrastructure and supply chain disruptions, and potential carbon taxation policies.

Studies released in May 2015 by the U.S. Global Change Research Program (“USGCRP”) showed the predicted effects of climate change and associated extreme weather events across North America in both high- and low-emissions scenarios. The study relied heavily on data culled from the Intergovernmental Panel on Climate Change (“IPCC”) and used “representative concentration pathways” (“RCPs”) to capture a range of plausible emission futures. The results, especially in the high-emissions scenarios, predict significant changes to temperature, timing of precipitation and seasonal runoff events in southwestern Idaho.

According to the studies, average annual temperatures in southwestern Idaho are projected to increase two to six degrees Fahrenheit by mid-century. Greater warming in the summer months will lead to an increase in the days with elevated heat index values across the state. Increased wildfire hazards are documented across the western U.S. due to climatic and other factors including increased fuel aridity, decreased forestry management, lengthened wildfire season and elevated fuel loads due to historical suppression. Wildfire risk to the DeLamar project includes direct risks to project infrastructure from rangeland fires, as well as indirect risks that include limiting access to site and worker safety/productivity issues.

Seasonally, climate models project increased precipitation in winter and decreased precipitation in summer leading to overall reduced soil moisture conditions across the region. Regionally, enhanced seasonality and increased rain-on-snow events are predicted to result in an increase in the amount of snowmelt runoff by the end of the century. Further analysis will be done to ensure that the project is engineered to weather these types of events and reduce risk associated with extreme flooding due to climate change related runoff or storms. Regulatory requirements for event containment and planning may also change depending on the interpretation of these models by the respective agencies.

26.0 RECOMMENDATIONS

Significant additional work is warranted for the DeLamar project based on the PFS results reproduced and included herein and the updated resources estimate summarized in this report. A program is recommended to allow for submitting a Mine Plan of Operations to the BLM by the end of 2023 and to advance the project to the feasibility level in 2024-2025. The estimated cost of the recommended program is $7.6 million.

Recommended work includes the optimization of mine planning, metallurgical recoveries, and infrastructure, the completion of permitting-related studies, and exploration and stockpile drilling.

The ongoing metallurgical testing should continue, with a focus on defining the metallurgical and mineralogical characteristics and variability for the mixed and non-oxide materials of the DeLamar area resources. Further work should also be done to optimize the concentrate regrind and leach parameters for the DeLamar area non-oxide material. Additionally, higher-grade silver mixed material from Florida Mountain should be tested for mill response. Evaluation of processing alternatives should continue to be examined for the DeLamar non-oxide materials, and optimization of processing scenarios should be continued for each of the major material types for the DeLamar and Florida Mountain resources. Further development of the geo-metallurgical model should continue to allow further optimization of ore routing to improve recoveries. Finally, metallurgical testing on production schedule-based composites will be needed as the project advances.

The newly estimated Indicated stockpile resources should be evaluated, with the goal of converting them into mineral reserves. As portions of the stockpiles have not been drilled as of the effective date of the resources and therefore not included in the current project resources, further drilling by sonic and/or RC (using the casing-advance system) is recommended to fully develop these stockpile resources.

There continues to be excellent potential to expand the extents of mineralization of economic interest within the DeLamar project, and the project therefore warrants additional investment. Following the completion of the feasibility study, drilling should be focused on both expanding the existing limits of the current project resources and testing targets peripheral to the resources. As a complement to the proposed drilling, an exploration program should be conducted that includes soil sampling and geological mapping to assist in identifying and refining drill targets. The continued collection of specific-gravity data from all of the proposed core drilling programs is highly recommended, especially within the current footprint of the project resources.

Estimated costs for the recommended work program outlined above are presented in Table 26-1. The estimated drilling costs are all-inclusive, as they include Integra's labor, drilling costs, access, drill-pad construction, assaying, etc., costs in addition to the contractor costs. It is the authors' opinion that the DeLamar project remains a project of merit that warrants the proposed program and associated level of expenditure.

Table 26-1 Integra Cost Estimate for the Recommended Work Program

Item	Estimated Cost US$
Exploration Drilling (2,000 meters)	$1,200,000
Stockpile Drilling (1,500 meters)	$600,000
Metallurgical Testwork	$400,000
Engineering, Design	$3,500,000
Permitting	$2,200,000
Total	***$7,600,000***

27.0 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.

Bond, 2020, *Summary of Integra Resources DeLamar Concentrate Testing Program*: Inter-Office Memo prepared for Hector Casapia, Jerritt Canyon Gold LLC.

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.

Botz, M.M. and J.O. Marsden, 2019, *Heap Leach Production Modeling: A Spreadsheet-Based Approach*: Journal Mining, Metallurgy & Exploration, v. 36, No. 6, p. 1041-1052.

BV Minerals, 2020, *Mineralogical Assessment of the Five Composites*: report prepared for McClelland Laboratories, Proposal No. PME1908311.

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.

DeLong, R.,2022a (February), untitled project communication document containing text for Section 4.4, received via email on February 11, 2022.

Delong, R., 2022b (February), untitled project communication document containing text for Section 20, received via email on February 25, 2022.

Dyer, T.D., Gustin M.M., Weiss, S.I., McPartland, J.S, Nopola J., Sletten, M., Bermudez, B., Ibrado, A.S., Gardner, J.F., Botz, M.M. and Welsh, J.D., 2022 (January), *Technical report and Preliminary Feasibility study for the Delamar and Florida Mountain gold silver project, Owyhee County, Idaho, USA*: NI 43-101 report prepared for Integra Resources Corp., 414 p.

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 Owyhee 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.

Enter, 2021, *Mineralogical and Textural Characterisation of Ore Samples from the DeLamar and Florida Mountain Deposit*: report prepared for Integra Resources Corp. by Vidence, Project References VL-20017, VL-21072, VL-210092, VL-21106.

Geldart, 2020, *An Investigation of Albion Process Atmospheric Oxidation of Two Concentrate Samples*: report prepared for McClelland Laboratories by SGS Canada Inc., Project 17955-01.

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.

Golder Associates Inc., 1989, *Geotechnical Services, Florida Mountain Project , 883-1507*: prepared by Golder Associates Inc., Seattle, WA, for Resource Associates of Alaska, Inc., Portland, OR.

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, 2019a (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.

Gustin M.M., Weiss, S.I., Dyer, T.D., McPartland, J.S, Woods, J.L., and Welsh, J.D., 2019b (October), *Technical Report and Preliminary Economic Assessment for the DeLamar and Florida Mountain Gold-Silver Project, Owyhee County, Idaho, USA*: NI 43-101 report prepared for Integra Resources Corp., 317 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.

Hill, 2019, *2019-SP-002 DeLamar Roast Data Summary*: internal memorandum prepared for M. McGlynn (Newmont).

Jordan, T., 2019 (March), *Sampling Narrative*: Internal memorandum prepared for DeLamar Mining Company.

Kappes, Cassiday & Associates, 2021 (April), *HPGR Preparation for MLI Project 4471*: memorandum prepared for McClelland Laboratories, Project No. 8839C / Report I.D. KCA0210022_MCC11_01.

Kilborn Engineering BC, 1988 (November), *Nerco Delamar Co. 'Floridan' Mountain Project – Review of Metallurgical Testwork*: report prepared for NERCO Minerals.

LaFronz, et al., 1989, *Heap Leach Facility Conceptual Design Development Report*, unpublished report for Nerco Minerals by Sergent Hauskins Beckwith Consulting Engineers, 200 pp.

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.

Marsden, J.O. and Botz, M.M., 2017, *Heap Leach Modeling – A Review of Approaches to Metal Production Forecasting:* Minerals & Metallurgical Processing, v. 34, No. 2, p. 53-64.

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.

McPartland, J.S., 2020a (March), *Memorandum on Metallurgical Testing - DeLamar Samples*: report prepared for Integra Resources Corp. by McClelland Laboratories, MLI Project No. 430720.

McPartland, J.S., 2020b (October), *Report on Heap Leach Cyanidation Testing on DeLamar Oxide and Transitional Material Type Samples*: report prepared for Integra Resources Corp. by McClelland Laboratories, MLI Project No. 4307.

McPartland, J.S., 2020c (October), *Report on Heap Leach Variability (Bottle Roll) Tests – Florida Mountain Deposit*: report prepared for Integra Resources Corp. by McClelland Laboratories, MLI Project No. 4471.

McPartland, J.S., 2021a (September), *Report on Heap Leach Testing – Florida Mountain Deposit*: report prepared for Integra Resources Corp. by McClelland Laboratories, MLI Project No. 4471.

McPartland, J.S., 2021b (September), *Summary Report on Mill Testing – Florida Mountain Non-Oxide Composites*: report prepared for Integra Resources Corp. by McClelland Laboratories, MLI Project No. 4471.

McPartland, J.S., 2021c (October), *Interim Report on Heap Leach Cyanidation Testing - DeLamar Oxide and Mixed Drill Core Composites*: MLI Job No. 4522.

McPartland, J.S., 2021d (October), *Report on Mill Testing – DeLamar Non-Oxide Drill Hole Composites*, report prepared for Integra Resources Corp. by McClelland Laboratories, MLI Job No. 4522.

McPartland, J.S., 2022 (January), *Summary Report – DeLamar and Florida Mountain PEA and PFS Metallurgical Testing*: report prepared for Integra Resources Corp. by McClelland Laboratories, MLI Job Nos. 4307, 4471 and 4522.

McPartland, J.S., 2023a (September), *Summary Report on Flotation Testing Program – DeLamar Variability and Optimization Composites* by McClelland Laboratories, MLI Job No. 4750.

McPartland, J.S., 2023b (October), *Memorandum – Update on DeLamar Waste Dump and Stockpile Drill Composite Bottle Roll Testing* by McClelland Laboratories, MLI Job No. 4876.

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.

NERCO DeLamar Company, 1987, *Rock Mechanics and Pit Slope Stability Study for the Glen Silver Pit, DeLamar Silver Mine, DeLamar, Idaho*: prepared by NERCO DeLamar Company, Portland, OR.

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.

Pocock Industrial, Inc., 2021 (December), *Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration and Pressure Filtration Studies*: report prepared for McClelland Laboratories, Project No. MLI 4471.

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.

Raffaldi, M, Haugen, B, and Nopola, J, 2021 (June), Geotechnical Prefeasibility Report: DeLamar/Florida Mountain Project, Idaho, United States Topical Report RSI-3115, unpublished report prepared for Integra Resources.

Raffaldi, M.J., Haugen, B.D., and Nopola, J.R., (2021), *Geotechnical Prefeasibility Report: DeLamar/Florida Mountain Project, Idaho, United States; Topical Report RSI-3115*: prepared by RESPEC for Integra Resources Corp., 159 p.

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.

Wickens, 2020 (December), *Summary Report on Metallurgical Test Results on Drill Hole and Bulk Samples from the DeLamar Project*: report prepared for Integra Resources Corp. by McClelland Laboratories, MLI Job No. 4307.

28.0 DATE AND SIGNATURE PAGE

Effective Date of report:	August 25, 2023
Completion Date of report:	October 31, 2023

"Thomas L. Dyer" Thomas L. Dyer, P.E.	***Date Signed:*** October 31, 2023
"Michael M. Gustin" Michael M. Gustin, C.P.G.	***Date Signed:*** October 31, 2023
"Jack S. McPartland" Jack S. McPartland, QP Member M.M.S.A.	***Date Signed:*** October 31, 2023
"John D. Welsh" John D. Welsh, P.E.	***Date Signed:*** October 31, 2023
"Matthew Sletten" Matthew Sletten, P.E.	***Date Signed:*** October 31, 2023
"Benjamin Bermudez" Benjamin Bermudez, P.E.	***Date Signed:*** October 31, 2023
"Jay Nopola" Jay Nopola, P.E.	***Date Signed:*** October 31, 2023
"John F. Gardner" John F. Gardner, P.E.	***Date Signed:*** October 31, 2023
"Michael M. Botz" Michael M. Botz, P.E.	***Date Signed:*** October 31, 2023

29.0 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 Principal Consultant by RESPEC Company LLC, 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 over 40 years and have extensive experience in precious-metal epithermal deposits, including the estimation of resources of such deposits in the western U.S. and Mexico. I am 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 am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
3. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("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.
4. I visited the DeLamar project site on August 16, 17, and 18, 2018, and on October 15, and October 27, 2020, October 19, 2022, and January 13, 2023.
5. I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.6, 1.12, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 20, 23, 24, 25, 26, 27, 28, and 29 of the Technical Report.
6. I was a co-author of previous technical reports prepared for Integra and assisted Kinross Gold Corporation with an evaluation of the project in 2016, but I have been and remain independent of Integra 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.
7. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible for contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
8. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 31st day of October 2023,

"Michael M. Gustin"

Michael M. Gustin, C.P.G.

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 32 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 am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
4. I am responsible for Sections 1.5 and 13 of the Technical Report
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of Integra applying all of the tests in section 1.5 of NI 43-101.
6. I visited the DeLamar project site on January 17, 2019. I am responsible for Item 1.5 and Item 13 of the Technical Report
7. I was a co-author of previous technical reports prepared for Integra in 2018 and 2022.
8. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
9. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 31st day of October 2023.

"Jack S. McPartland"

Signature of Qualified Person

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 am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
3. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and 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.
4. I last visited the property on October 27, 2020.
5. I am responsible for Section 17.3.3, 18.2, 18.3, 18.4, and portions of 21 and 25 of the Technical Report.
6. I am independent of Integra 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.
7. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
8. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 31st day of October 2023.

John D. Welsh

John D. Welsh, P.E.

CERTIFICATE OF QUALIFIED PERSON

MATTHEW SLETTEN, P.E.

I, Matthew Sletten, PE, do hereby certify that:

1. I am a Project Manager of M3 Engineering & Technology Corp., 2175 W. Pecos Rd. Suite 3, Chandler, AZ 85224
2. I graduated with a BS in Civil Engineering and an MS in Civil Engineering from the South Dakota School of Mines and Technology in 2004 and 2006, respectively.
3. I am a Professional Engineer in good standing in the State of Arizona in the area of Civil Engineering.
4. I have worked as an engineer and project manager in the base metals and precious metals industry for a total of 17 years. My experience includes detailed engineering, engineering management, project management, corporate management, capital and operating cost development and report development for major mining projects throughout the world.
5. I am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
6. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am responsible for the preparation of Section 18.7 and 18.8. I visited the project site on October 27th, 2020.
8. I have no prior involvement with the project or property that is the subject of the Technical Report.
9. I am independent of Integra 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.
10. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
11. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 31st day of October 2023.

(Signed) "*Matthew Sletten*"
Signature of Qualified Person

Matthew Sletten, PE

CERTIFICATE OF QUALIFIED PERSON

THOMAS L. DYER, P.E.

I, Thomas L. Dyer, P.E., do hereby certify that I am currently employed as Principal Consultant by RESPEC Company LLC, 210 South Rock Blvd., Reno, Nevada 89502, and:

1. 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.
2. I have worked as a mining engineer for more than 27 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.
3. I am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
4. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited Florida Mountain and DeLamar properties on October 27, 2020 where I inspected the property, current facilities, and access to the deposits.
6. I take responsibility for Sections 15, 16, 18, 19, 21 (except for 21.2 and 21.6), and 22 of this Technical Report, subject to those issues discussed in Section 3. I take joint responsibility for Sections 1, 24, 25 and 26.
7. I was a co-author of previous technical reports prepared for Integra, but I have been and remain independent of Integra 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. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
8. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 31st day of October 2023.

"Thomas L. Dyer"

Signature of Qualified Person

CERTIFICATE OF QUALIFIED PERSONS

JOHN F. GARDNER, PH.D., P.E.

I, John F. Gardner, Ph.D., P.E., do hereby certify that:

1. I am currently on contract as Senior Engineering Advisor with Warm Springs Consulting at 217 South 11^{th} St, Boise, ID, 83702.
2. I earned a Bachelor of Science degree in Mechanical Engineering from Cleveland State University in 1981, a Masters of Science degree in Mechanical Engineering from the Ohio State University in 1983 and a PhD in Mechanical Engineering from the Ohio State University in 1987. I am a Registered Professional Engineer in the state of Idaho (#P-9912) and a Fellow of the American Society of Mechanical Engineers.
3. I worked as an engineering professor for 34 years, 13 years at Penn State (main campus) and the remainder at Boise State University. During that time, I performed research and taught in various areas related to dynamic systems and controls. For the past 10 years, my work has been in renewable energy resource evaluation and modeling of smart grid and microgrid applications.
4. I am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have not visited Florida Mountain or DeLamar.
7. I take responsibility for the content of section 18.5 of the Technical Report.
8. I am independent of Integra 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.
9. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
11. I have had no prior involvement with the property that is the subject of this report.

Dated this 31st day of October 2023.

"John F Gardner"

Signature of Qualified Person

CERTIFICATE OF QUALIFIED PERSONS

MICHAEL M. BOTZ, P.E.

I, Michael M. Botz, P.E., do hereby certify that:

1. I am currently employed by Elbow Creek Engineering Inc. (5825 Lazy Lane, Billings, MT 59106 USA) as a Consulting Process Engineer and President.
2. I am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
3. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4. I graduated with a Bachelor of Science degree in Chemical Engineering from Montana State University (Bozeman, MT) in 1991 and graduated with a Master of Science degree in Chemical Engineering from Purdue University (West Lafayette, IN) in 1993.
5. I have worked as a process engineer for more than 28 years since my graduation. Much of my consulting experience is with precious metals metallurgical operations in the USA, Latin America and other countries worldwide. This includes preparing precious metal production forecasts for heap leach operations and publishing peer-reviewed journal articles on the subject.
6. I am a registered Professional Engineer in the states of Colorado (0047940), Montana (10411PE), Nevada (022431) Washington (21022737) and Wyoming (PE 9489).
7. I have not visited or inspected the properties that are the subject of this Technical Report.
8. I have had no prior involvement with the project or properties that are the subject of this Technical Report.
9. I am responsible for the preparation of portions of Sections 17.3.2 and 17.3.4 of this Technical Report.
10. I am independent of Integra 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.
11. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
12. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 31st day of October 2023.

Michael M. Botz, P.E.

Signature of Qualified Person

CERTIFICATE OF QUALIFIED PERSONS

BENJAMIN BERMUDEZ, P.E.

I, Benjamin Bermudez, PE, do hereby certify that:

1. I am currently employed as a Chemical/Process Engineer at M3 Engineering & Technology Corporation, M3 Engineering & Technology Corp., 2175 W. Pecos Rd. Suite 3, Chandler, AZ 85224.
2. I am a graduate of Arizona State University and received a Bachelor of Science degree in Chemical Engineering in 2009.
3. I am a registered professional engineer in good standing in the State of Arizona in the area of Chemical Engineering (No. 54919).
4. I have worked as an engineer for a total of 15 years. My experience includes mineral process plant engineering, support of new and on-going process plant operations, financial modeling of mineral properties, and project management.
5. I am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
6. I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am responsible for the preparation of Section 1.9, 17 (except 17.3.3, 17.3.4), 21.2 (except 21.2.6 and 21.2.7), and 21.6. I have not visited the project site.
8. I have no prior involvement with the project or property that is the subject of the Technical Report.
9. I am independent of Integra 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.
10. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
11. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 31st day of October 2023.

(Signed) ***"Benjamin Bermudez"***
Signature of Qualified Person

Benjamin Bermudez, P.E.

CERTIFICATE OF QUALIFIED PERSONS

JAY R. NOPOLA, P.E.

I, Jay R. Nopola, P.E., P.Eng, C.P.G., do hereby certify that:

1. I am currently employed as Principal Consultant at RESPEC, whose address is 3824 Jet Dr., Rapid City, SD. I am a contributing author of the technical report titled "Technical Report for the DeLamar and Florida Mountain Gold – Silver Project, Owyhee County, Idaho, USA" dated October 31, 2023 with an effective date of August 25, 2023 (the "Technical Report") prepared for Integra Resources Corp. ("Integra").
2. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
3. I take responsibility for Section 15.1 of this Technical Report.
4. I graduated with a Bachelor of Science degree in Geological Engineering from South Dakota School of Mines and Technology in 2001 and with a Masters of Science degree in Geological Engineering from South Dakota School of Mines and Technology in 2013. I am a Registered Professional Engineer in the state of South Dakota (#10118), a Registered Professional Engineer in the province of Ontario (#100560907), and a Certified Professional Geologist (#11502) with the American Institute of Professional Geologists. I have worked in the field of geological engineering for 20 years with a focus on rock mechanics.
5. I have visited Florida Mountain and DeLamar properties on September 24, 2020 where I inspected the existing pits and reviewed available core.
6. I am independent of Integra 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.
7. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report not misleading.
8. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
9. I have had no prior involvement with the property that is the subject of this Technical Report.

Dated this 31st day of October 2023.

"Jay R. Nopola"

Signature of Qualified Person
Jay R. Nopola, P.E.

MINE DEVELOPMENT ASSOCIATES

A Division of RESPEC

APPENDIX A

LISTING OF PATENTED AND UNPATENTED FEDERAL MINING CLAIMS AND LEASED LAND

775-856-5700

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

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

G. Fewkes Trust

Description: The following 7 patented claims in Sec 6, T5S,R3W, BM & Sec 1, T5S,R4W, BM

Royalty: 2% NSR

Survey Number	Patent Number	Claim Name
MS 3182	1036006	Idaho Lode

MS 1096	30108	Lone Tree Lode
MS 1114	1036007	Crown Point Lode
MS 1104A	26609	Pluto Lode
MS 3181	1036005	American Eagle Lode
MS 3181	1036005	Long Gulch Lode
MS 1258B	30220	JM Guffy Millsite

H. Green

Description: The following 4 unpatented placer claims in Section 12 T 5SR4W, BM; Section 7 T5S,R3W, BM

Royalty:

Claim Name: Monarch 1-4 IMC #05267468 - 71 County Inst # 309982-85

Leased Lands (Owyhee County, ID):

Seven State of Idaho Department of Lands Leases

	Lease No.	Acreage	Description	Status
1	E600067	396	T4S,R5W,S.36	Issued
2	E600085	640	T4S,R4W,S.36	Issued
3	E600086	601	T4S,R4W,S.35; T5S,R5W,1	Issued
4	E600090	640	T5S,R4W,S.16	Issued
5	E600092	514	T4S,R4W,S.31	Issued
6	E600093	557	T4S,R4W,S.28,29,33; T5S,R4W,S7	Issued
7	E600091	510	T4S,R4W,S.30	Issued

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

Claim #	Claim Name	BLM #
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
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

Claim #	Claim Name	BLM #
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
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

Claim #	Claim Name	BLM #
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
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

Claim #	Claim Name	BLM #
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
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

Claim #	Claim Name	BLM #
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
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

Claim #	Claim Name	BLM #
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
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

Claim #	Claim Name	BLM #
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

Part 6	BG 15-24; 27-46	IMC 232210-219; 232222-241
	DD 1-8	IMC 226002-009