Revised and Amended Initial Assessment

Chandalar Mine, Chandalar Gold District

Alaska U.S.A.

Revised and Amended Date:

February 24, 2023

Effective Date: May 31, 2021

Prepared for:

Goldrich Mining Company

Prepared by:

Global Resource Engineering, Ltd.

Qualified Persons:

Hamid Samari, PhD, QP, MMSA 01519QP, Senior Geologist, GRE

J. Todd Harvey, PhD, QP, SME 4144120RM, President and Director of Process Engineering, GRE

Terre Lane, QP, MMSA 01407QP, SME 4053005RM, Principal Mining Engineer, GRE

Richard Hughes, PE (AK EM-5531) Mining Engineer, Goldrich

Larry Breckenridge, PE (CO 38048) Principal Environmental Engineer, GRE

Date and Signature Page

This Technical Report on the Chandalar Project is submitted to Goldrich Mining Company and is effective May 31, 2021, revised and amended February 24, 2023.

The Qualified Persons and Responsible Report Sections follow:

Qualified Person	Responsible for Report Sections
Hamid Samari, PhD	All of 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 7, 8, 9, 10, 11, and 12
J. Todd Harvey, PhD	All of 1.10, 1.13, 13, and 17
Terre Lane, MMSA QP, registered member SME	All of 1.1 - 1.3, 1.11, 1.12 1.14, 1.15, 1.18, 1.19, 1.20, 2, 3, 4.1 - 4.4, 5, 6, 14, 15, 16, 18, 19, and 21 - 27
Richard Hughes, PE	
Larry Breckenridge, PE	All of 1.16, 1.17, 4.5, 4.6, and 20

(Signed) "Hamid Samari" 2/24/2023
Signature Hamid Samari Date

(Signed) "J. Todd Harvey" 2/24/2023
Signature J. Todd Harvey Date

(Signed) "Terre Lane" 2/24/2023
Signature Terre Lane

(Signed) "Richard Hughes" 2/24/2023
Signature Richard Hughes

(Signed) "J. Larry Breckenridge" 2/24/2023
Signature J. Larry Breckenridge

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF PHOTOS

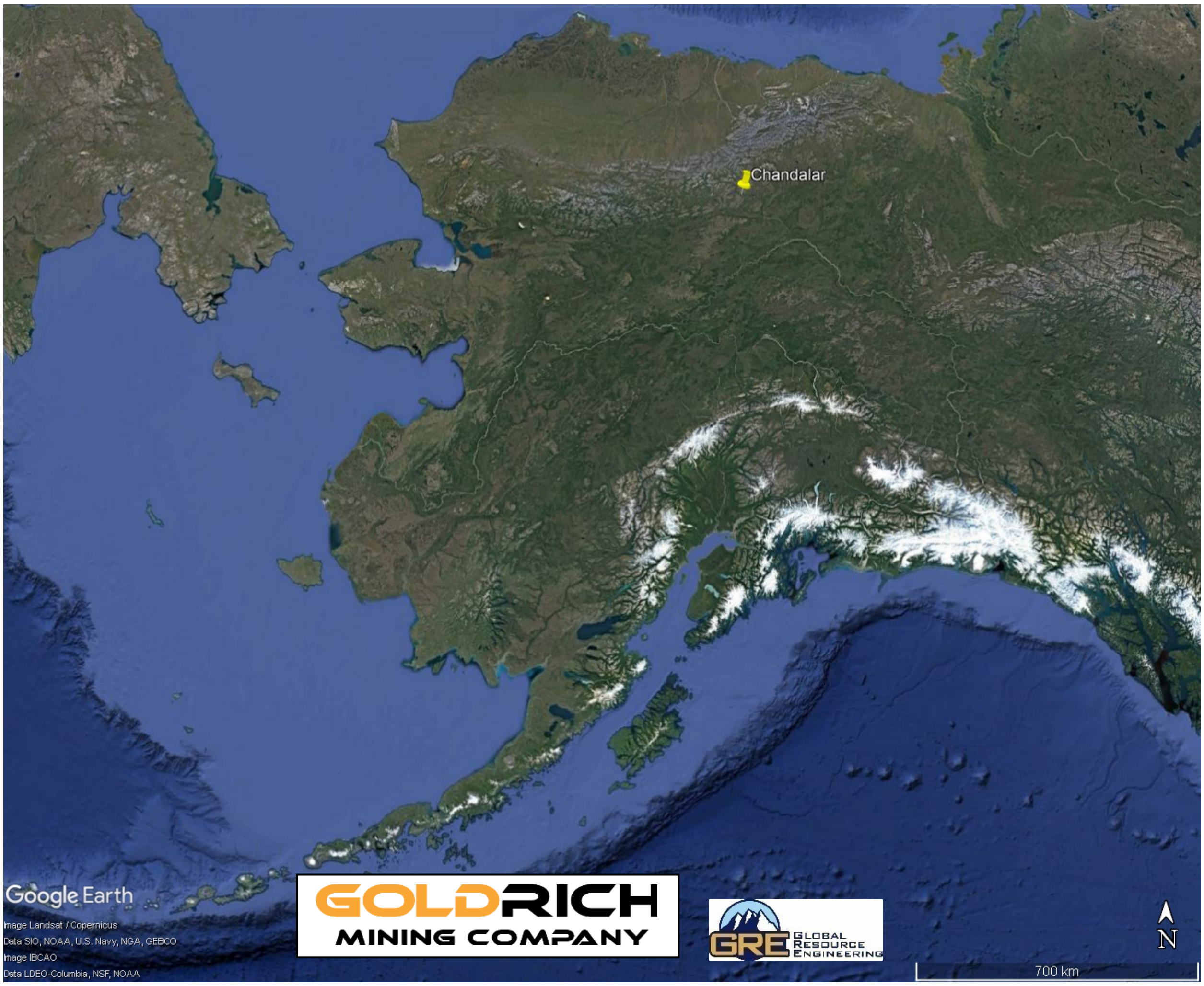
ABBREVIATIONS AND ACRONYMS

ACOE	US Army Corps of Engineers
ADEC	Alaska Department of Environmental Conservation
ADNR	Alaska Department of Natural Resources
APMA	Alaska Placer Mining Authorization
Au	gold
bcy	bank cubic yards
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CSAMT	Controlled-source Audio-Frequency Magnetotellurics
GNP	Goldrich NyacAU Placer, LLC
Goldrich	Goldrich Mining Company
gpm	gallons per minute
GRE	Global Resource Engineering, Ltd.
GPS	global positioning system
ha	hectare
hp	horsepower
IA	Initial Assessment
IRR	internal rate of return
km	kilometer
LOC	Line of Credit
LSGMC	Little Squaw Gold Mining Company
Ma	million years
mg	milligrams
MTRSC	Meridian-Township-Range-Section-Claim
NAD	North American Datum
NI	National Instrument
NPV	net present value
opy	ounces per cubic yard
oz	troy ounce
oz/bcy	ounces per bank cubic yard
IA	Initial Assessment
ppm	parts per million
QP	Qualified Person
Raw Gold	Gold as recovered from the placer deposit, historically 84% gold and 16% other metals like silver and copper (referred to as 840 fine)
RC	reverse circulation
RDi	Resource Development Inc.
UTM	Universal Trans Mercator
ZTEM	z-Axis Tipper electromagnetic System

1.0 Executive Summary

Goldrich Mining Company (Goldrich) is a U.S. based mineral resource company focused on developing the Chandalar gold district in Alaska, USA. Goldrich has retained Global Resource Engineering, Ltd. (GRE) to prepare this Initial Assessment (IA) for the Chandalar deposit within the greater Chandalar Gold Project area, located along the southern flank of the Brooks Range in north east Alaska (Figure 1-1).

Figure 1-1: Chandalar Project Location



This report presents the results of the IA based on data collected from 2007 through 2018, including a mineral resource estimate updated to incorporate drilling completed in 2013 and 2017, as well as mining completed from 2015 through 2018. This report is intended to fulfill 17 Code of Federal Regulations (CFR) §229, "*Standard Instructions for Filing Forms Under Securities Act of 1933, Securities Exchange Act of 1934 and Energy Policy and Conservation Act of 1975 – Regulation S-K*," subsection 1300, "*Disclosure by Registrants Engaged in Mining Operations*." The mineral resource estimate presented herein is classified according to 17 CFR §229.1300 – (Item 1300) Definitions.

This IA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under 17 CFR §229.1300 – (Item 1300). Readers are advised that there is no certainty that the results projected in this IA will be realized.

The Qualified Persons (QPs) responsible for the preparation of this Technical Report are:

- Hamid Samari, PhD
- J. Todd Harvey, PhD
- Terre A. Lane
- Richard Hughes, PE
- Larry Breckenridge, PE

1.1 Property Description and Location

The Chandalar Gold Project is located in north-central Alaska, north of the Arctic circle, on Alaska state land surface along the southern flank of the Brooks Range. The property is situated at approximately 67°33'N latitude and 148°10'W longitude, roughly 190 air miles (306 kilometers [km]) north of Fairbanks and 48 air miles (77 km) east-northeast of the community of Coldfoot.

Goldrich (formerly the Little Squaw Gold Mining Company) holds approximately 23,000 acres (9,300 hectares [ha]) of minerals rights covering much of the Chandalar Mining District. The property consists of 23 patented federal mining claims and 197 Traditional and MTRSC 40-acre State of Alaska unpatented mining claims. The claim block covers all or portions of:

- Sections 1, 12 and 13 of R4W, T31N
- Sections 1-11, 15, 16 and 21 of R3W, T31N
- Sections 4-6 of R2W, T31N
- Sections 22-28 and 31-36 of R3W, T32N
- Sections 28-33 of R2W, T32N

1.1.1 Arbitration

Prior to August 8, 2019, the Chandalar mine surface placer deposit was leased by Goldrich NyacAU Placer, LLC (GNP), a 50/50% joint venture between Goldrich and NyacAU, to mine the various placer deposits that occur throughout Goldrich’s 17,100-acre Chandalar gold project in Alaska. NyacAU was the manager of the joint venture.

On August 8, 2019, Goldrich announced that the joint venture was dissolved because GNP failed to meet the Minimum Production Requirements. According to the terms of the joint venture operating agreement, GNP was required to pay a contractual Minimum Production Requirement of 1,100 ounces for 2016, 1,200 ounces for 2017, and 1,300 ounces for 2018 to Goldrich by October 31, 2018. The Minimum Production Requirement for each year was determined based on the spot price of gold on December 1 of the preceding year.

NyacAU filed the formal Notice of Dissolution in May 2019 and received the certificate of dissolution in July with an effective date of June 3, 2019. GNP's lease to mine the placer properties terminated upon dissolution of GNP, and GNP has no further rights to mine the placer properties located on Goldrich's mining claims. The claims remain the property of Goldrich.

GNP is now in the liquidation process, and NyacAU, as the manager of GNP, shall act as liquidator to wind up the joint venture. Most of the equipment used by the joint venture has already been moved off the mine site. The joint venture parties agreed to the Panel retaining jurisdiction and oversight over the liquidation process to its conclusion. Further details of matters relating to the mining claims and the arbitration are discussed in Section 4.4.

1.1.2 Environmental Liability

The Chandalar project includes several sites of historical mining activity, including both placer gold operations along local creek valleys and lode gold mines in higher elevations. There are few non-reclaimed placer tailings remaining at surface, and no outstanding reclamation is required in those areas. Tailings impoundment ponds from a permitted load mine and milling operation in the late 1980s were sealed in the early 1990s with approval of the Alaska Department of Environmental Conservation (ADEC).

Several minor issues related to historical lode gold mining and processing activities, specifically at the Tobin Creek mill site, were identified in the 1980s by the ADEC. The property lessee declared bankruptcy soon thereafter and did not perform the clean-up; Goldrich carries a current $100,000 accrued remediation cost to execute the approved plan.

Mr. Breckenridge completed a first principal estimate for reclamation of the Chandalar mining area disturbed by GNP's mining activities to the US Army Corps of Engineers (ACOE) permit-approved reclamation plan, which included backfilling all waste into the existing mining pits and re-establishing pre-mining ground contours. The estimate showed reclamation costs of approximately $18.0 million. GRE has proposed a new reclamation plan that will include concurrent backfilling of the new pits to new contours that would include ponds and would not bring the pits to pre-mining contours. In addition, some waste material would remain on waste piles. The costs and work to complete the new reclamation plan are included in this IA and the results reported are net of all costs for reclamation.

1.2 Accessibility, Climate, Infrastructure, and Physiography

Air access to the project area is available year-round via aircraft from Fairbanks to Goldrich's 5,000-foot (1,520-meter) airstrip at Squaw Lake. Overland access is available only during the winter season and is provided by a 78-mile-long (126-km-long) ice road from Coldfoot to the state-owned airport at Chandalar Lake. From Chandalar Lake, the project is accessed either by a 17-mile (27-km) winter dozer trail to Goldrich's airstrip and camp at Squaw Lake or by a 7-mile (11-km) pioneer road along Tobin Creek to the Tobin Creek mill and mine camp. The Squaw Lake and Tobin Creek camps are connected to all major prospects in the district via a 27.5-mile (44-km) network of all-season mine roads.

Local terrain is rugged and steep, consisting of talus-covered hill slopes and deeply incised, alluvium-filled valleys. The higher elevations are barren of vegetation except for moss, lichen, and some grasses, and the lower country is mantled by relatively continuous spruce forests. Permafrost in the area is continuous and extends to depths of several hundred feet.

Snowmelt generally occurs toward the end of May and is followed by an intensive 90-day growing season with more than 20 hours of daylight and daytime temperatures ranging from 60 to 80°F (15.6 to 26.7°C). Freezing temperatures return in late August, with freeze-up typically occurring by early- to mid-October. Winter temperatures, particularly in the lower elevations, can drop to -50°F (-45.6°C) or colder for extended periods. Annual precipitation is 15 to 20 inches (38 to 51 centimeters), occurring mostly as late summer rain and early winter snowfall.

Infrastructure includes a 5,000-foot (1.5 km) airstrip and adjoining 25-person camp at Squaw Lake and a 27.5-mile (44-km) network of mine roads. The mine roads provide all-season access to all major gold prospects in the project area and to older camps and airstrips at the Tobin Creek mill site and Big Creek placer site. The Squaw Lake airstrip can readily handle multi-engine cargo aircraft up to C-130 size. The adjoining camp consists of various equipment repair facilities, storage units, office buildings, and sleeping and kitchen facilities.

As there is no electrical grid in northern Alaska, the project relies on diesel generators to power operations.

Goldrich maintains a State of Alaska water right, issued in 1985, allowing the withdrawal of up to 3,000 gallons (11,360 liters) of water per minute for placer mining, and 72,000 gallons (272,550 liters) per day for lode mining. The water can be withdrawn from any of the local streams specified in the permit for use from April through October. The water right is maintained by paying an annual $50 administrative fee and by demonstrating some beneficial use of the water at least once in any 5-year period.

Potable water is sourced from a natural spring located on a patented 5-acre (2-ha) mill site claim at the Squaw Lake camp and airstrip site. The spring flows 140 gallons (530 liters) per minute (gpm) at a year around temperature of 40°F (4.4°C). Water for gravel washing and sluicing is obtained from the Big Squaw Creek.

Goldrich surface rights and land holdings are sufficient for all planned operations and placer and lode gold exploration and production within the Chandalar Gold project area.

1.3 History

The Chandalar district has gone through numerous owners beginning in 1909 up to the present-day ownership by Goldrich.

The long and storied history of prospecting and mining in the Chandalar district is described in detail in several reports: Wolff (1997); Barker and Bundtzen (2004); Strandberg (1990); Barker et.al. (2009). Rich gold placer deposits were first discovered in the district in 1905 along Little Squaw Creek, within the present day Chandalar project area, which was followed by a flurry of prospecting activity, ultimately leading to the discovery of bedrock lode deposits. Maddren (1910) reports that by 1909, four principal auriferous quartz veins had been identified. To date, more than 30 lode deposits and historic lode prospects are known to occur within the Chandalar project area alone.

Placer gold discoveries in the Chandalar district were relatively deep under frozen overburden. By 1916, shallow, open-cut placer gold mines were playing out and attention shifted to developing placer drift mines (underground operations). Most notable was the Little Squaw Bench, including the Mello Bench, where about 30,000 troy ounces (oz) of gold (Au) were reportedly recovered from gravel averaging 0.96 oz Au/cubic yard (Strandberg, Jr., 1990). The Mello Bench is located on the lower portion of Little Squaw Creek in the southwest quarter of Section 26, T32N, R3W of the Fairbanks Meridian. Drift mining continued through the 1920s but declined in the 1930s as the remaining ground was deeper or lower grade, or in many cases not frozen.

Mining and exploration activity have been carried out in the Chandalar project area sporadically since 1967, when the Tobin Creek placer mine was expanded by the Chandalar Gold Mining and Milling Company. A 100 ton per day (tpd) crush/grind/gravity mill was constructed in 1969, and further exploration, including drilling on both lode and placer targets, was completed throughout the property by a variety of lease holders through 2003, when Goldrich (as LSCMG) acquired the project.

Total documented historical gold production from the entire Chandalar Mining District is approximately 112,000 troy ounces gold, mostly placer gold (Walters, 2020). In 2007, Goldrich discovered and partially drilled out a large placer gold deposit in the Little Squaw Creek drainage, and in 2009, opened the Chandalar Mine as a test project. Favorable results led to the expansion of the mine in 2010. Total production from 2009 to 2018 was approximately 44,209 ounces of fine gold.

1.4 Geological Setting and Mineralization

The Chandalar district is largely or entirely underlain by the Coldfoot subterrane, which consists mainly of Proterozoic to Lower Paleozoic metasedimentary schist intruded and overlain by bimodal metavolcanics and granitic rocks of Devonian age. In the Chandalar district, metamorphic rocks of the Coldfoot subterrane include schist, phyllite, and slate, with minor amounts of meta-gabbro and meta-diabase. The metamorphic rocks in the Chandalar project area are divided into Upper Plate (Mikado phyllite, Quartz-chlorite-muscovite schist, light grey, blocky, quartz-rich, muscovite-oligoclase schist) and Lower Plate (black schist, phyllite, slate, and quartzite) sequences that are separated by a major low-angle thrust fault plane. (Barker, et al., 2009; Mendham, et al., 2018)

Prior to the Pleistocene glacial advances, ground surface in the project area was low relief. Ancestral drainages, like Little Squaw Creek, were immature second-order streams that formed relatively large alluvial fans on the base level lowland to the north. Pre-glacial surficial features were buried under ice and, ultimately, lateral moraines and meltwater silt, clay, and glaciofluvial marginal deposits. There, the sedimentary section can be divided into a barren upper glacial till section and a lower gold-bearing fluvial section. Locally, the contact between the two is sharp, but typically there is a mixed zone between the two and, overall, the contact is gradational. (Barker, et al., 2009; Mendham, et al., 2018)

Lode mineralization has been identified in veins, one with an exposed length of 150 feet and with widths from 2 to 10 feet. Most vein systems are closely situated within or in the adjacent hanging wall of the major shear faults, but there are exceptions. The Chandalar vein systems sort into two groups. The first group (e.g., Mikado, Eneveloe, Pioneer) is discontinuously mineralized major quartz veins also associated with subparallel gold-bearing lenses, parallel veins, and stringer and sheeted zones, within enveloping zones of alteration, shearing, and gouge. The second group of vein systems (e.g., the Little Squaw, Crystal, Grubstake East and West, the Jackpot, and perhaps the Star) is found in east-west-trending fractures in relatively close proximity to the major shear zones, but more distal than those described above. This type of deposit occurs in subparallel splay faults or fractures that horsetail off the major shear zones. This series of gold-bearing quartz veins-fault zones are likely the source of the gold within the placer deposits in Little Creek catchment. (Barker, et al., 2009; Mendham, et al., 2018)

1.5 Deposit Types

Lode gold occurs throughout the Chandalar property in definable systems of veins, veinlets, disseminations, and auriferous lenses of quartz within or adjacent to northwest-trending shear zones. Lode gold deposits of the Chandalar mining district were previously classified as low-sulfide, quartz-sulfide-gold epithermal vein deposits (Ashworth, 1983). However, Bolin (1984) and Rose et al. (1988) concluded that evidence of boiling did not exist and that the veins were apparently mesothermal, implying deep-seated mineralized systems. A 2004 data review for LSGMC (Barker, et al., 2004) concurred with a mesothermal classification. Data from four exploration seasons support interpretation of the Chandalar veins as metasediment-hosted, orogenic, low-sulfide mesothermal deposits.

Placer gold in the Chandalar Mining district was liberated from lode sources of the former highland weathering surface. The Little Squaw Creek is a second-order stream with valuable gold deposits concentrated in placers in its upper reach and in a wide alluvial fan placer where it exits the canyon. This deposit is geologically characterized as an aggradational placer gold deposit. It is unusual in the sense that it is the only such known alluvial, or placer, gold deposit in Alaska, although many exist in Siberia.

1.6 Exploration

Other than drilling, exploration activities carried out by Goldrich include: local mapping of about 40 identified prospect areas; collection and geochemical analyses of approximately 1,350 soil, 1,305 rock, 67 stream sediment, and 11 water samples; preparation of anomaly maps; a trenching program of 45 trenches with collection of about 550 trench-wall channel samples; and ground magnetometer survey grids of 15 prospect areas and survey lines totaling 28 miles.

To date, Goldrich has collected and assayed a total of 3,431 surface samples covering approximately 65% of Goldrich's property and analyzed approximately 4,500 drill samples.

Two airborne geophysical surveys have also been completed. In 2011, approximately 770 line-miles (1,246 line-kilometers) were flown by an international geophysical contractor over the entire Chandalar property along flight lines 100 meters apart. In 2014, Goldrich completed another airborne radiometric and magnetic survey, also approximately 770 line-miles (1,234 line-kilometers) to test an intrusion-related model for emplacement of lode quartz-gold occurrences.

1.7 Drilling

In 2006, Goldrich conducted a 7,763-foot reverse circulation, 39-hole reconnaissance-level lode exploration drill program on nine of some thirty gold load prospects.

In 2007, Goldrich conducted a 15,304-foot, 107-hole reverse circulation (RC) placer evaluation drill program totaling 14,856.5 feet. Of the 107 holes collared, 87 were completed to bedrock, and 20 holes were either abandoned due to ground caving or swelling or were terminated at the full extent (210 feet) of the available drill rod without reaching bedrock.

In 2011, Goldrich completed a 25-hole, 14,444-foot (4,404-meter) exploratory program, using HQ size core, tested six prospect areas located along a 4-km (2.5-mile) long northeast trending belt of gold showings.

Subsequent to GNP leasing the placer claims from Goldrich, in 2013, 72 sonic dill holes totaling 6,253.7 feet were completed and in 2017, 230 sonic drill holes totaling 14,303 feet were drilled.

1.8 Sampling Preparation, Analyses, and Security

Pan and trench sampling were performed as early-phase exploration to collect sufficient information on which to base future drilling. Drilling on placer targets was performed in 2007, 2013, and 2017, and the collective results were used to support the mineral resource estimate.

Sample preparation included the following:

- Samples were collected from the surface to bedrock every five feet in 2007 and 2013, and every two feet in 2017.
- Typically, two to three buckets of placer material were collected per sample interval. The volume of each sample was measured with a measuring stick, recorded in the inventory notebook, and re-entered into the field panning log.
- Standard panning techniques, including double panning, reduced the sample to a high-grade pan concentrate. The pan concentrate was rinsed into a labelled plastic bag with a zip-lock closure.
- Placer sample concentrates, stored in U.S. Army ammunition cans, were placed into a heavy-duty aluminum strong box in the drill geologist’s office. Placer concentrates received in the Fairbanks laboratory were immediately entered into the laboratory’s Sample Inventory Log to check for missing sample intervals.
- Samples were then organized and double-panned into a tub in which all pan tailings were accumulated (composited) for a discreet drill hole.
- A high-grade pan concentrate was produced, from which all visible gold was extracted using a pipette. The gold was transferred to a labelled glass vial for further analysis.
- The remaining pan concentrates were transferred to an aluminum weighing boat and dried over low temperature heat. When dry, the heavy minerals were described by relative abundance with the aid of a binocular microscope.
- The pan concentrate was weighed and placed in a labelled paper envelope for storage.
- The accumulated pan tailings for each hole were dried and saved in a labelled, sealed plastic bag.

- In skilled hands, a nearly clean separation of gold from gangue can be made, and the resultant fractions were dried in the finishing pan over low-temperature heat.
- Gold particles were classified visually into five weight categories in order to conduct a color count.
- A digital image of each sample with visible gold was captured, edited, and saved on electronic media.
- The sample was then weighed with an Ohaus™ digital analytical balance. Nuggets were weighed and recorded individually.
- All the sample vials for a hole were bundled in a labelled cloth sample bag.
- The gold particles (vials), pan concentrates (paper envelopes), and pan tailings (plastic bag) were then placed in a custom, high strength, labelled cardboard box for storage. These smaller boxes were then placed into larger, heavy-duty labelled cardboard boxes, organized by drill line.
- Finally, all quantitative and descriptive data were entered onto the Weigh-Up Log. A unique label was created at every stage of sample transfer so that the integrity of sample identity was secure.

1.9 Data Validation

GRE QPs Dr. Samari and Ms. Lane visited the property on September 4 and 5, 2019. In addition to visiting the Chandalar placer area, the party toured all existing facilities and observed all equipment remaining from previous phases of exploration and mining activities. Drill hole collars had been obliterated by mining activity, making it impossible to confirm drill hole locations. Instead, GRE checked the approximate locations of the drilling lines at Chandalar from line 1.2 in the north to line 10 in the south.

GRE's QPs collected five 5-gallon buckets from the lower gold bearing fluvial section of the Chandalar deposit for panning at the site. The QPs also selected two chip rocks samples for assay analysis. Only one of the five bucket samples had a gold bead, which weighed 28.35 mg. The assay results from two chip samples were negative and did not show gold. The other sample, from quartz vein in schist, showed that not all quartz veins are gold-bearing quartz veins in the Little Squaw catchment.

Field observations made during the site visit generally confirm previous reports and maps of the geology of the project area. Based on the results of GRE's review of the sampling effort, verification of drilling lines and their profiles in the field, gold grade check sample analysis, the results of the QP's on-site sampling and panning, and the results of both manual and mechanical database audit efforts, the QP considers the gold grade data contained in the project database to be reasonably accurate and suitable for use in estimating mineral resources.

1.10 Mineral Processing and Metallurgical Testing

Chandalar project is a placer deposit that uses gravity recovery. Any gold contained within the rock matrix of the gravel that is not recovered in a gravity concentrate is not included in the estimation of head grade or incorporated in the recovery. The gold recovered in the gravity concentrate is the recoverable gold, i.e., gold recovery of the concentrate is 100%. Historical mining validates this observation.

Mining operations in 2009 and 2010 at Chandalar employed a traditional wash plant consisting of a feed hopper with water sprays to move the material into a rotating trommel, after which the undersize reported to a series of sluice boxes fitted with riffles and miner's moss. Mining operations from 2013 through 2018 used a feed hopper and screens with water sprays instead of a trommel. The process method, gravity concentration, recovers gold particles that have been liberated from the host rock. The sampling method similarly recovers the liberated gold particles, thus gold recovery during operations is estimated to be 100 percent of the sampled grade. Historical refinery records show the gold from the Chandalar placer are 84% pure (840 fine); GRE applied the 84% purity factor in the economic analysis.

1.11 Mineral Resource Estimate

The Chandalar resource database includes 395 drill holes, totaling 35,930 feet (104 2007 drill holes totaling 15,373.4 feet, 61 2013 drill holes totaling 6,253.7 feet, and 230 2017 drill holes totaling 14,303 feet). The Goldrich-provided database contained several grade fields, including one named "Grade_HL" and another named "composite au 2013,2017." The 2007 and 2013 drilling samples (5,042 samples) were analyzed by Goldrich (contained in the Grade_HL field) and 7,671 (contained in the composite au 2013,2017 field) were analyzed by NyacAU, of which 5,632 were used, resulting in 10,674 assay values in the complete data set.

Gold grades from the 2007 campaign were provided as 999 fine gold values. Grades for the 2013 and 2017 drilling were provided in raw gold with an 870 fineness. GRE converted the 2007 values to raw gold (870 fineness) to create a consistent data set of raw gold for the resource estimation. Drill intervals with missing assay values were set to 0 under the assumption that they were deemed unmineralized intervals by the geologist during the sample selection.

A Goldrich consulting geologist, Cherrie Lam, created a pay gravel wire frame solid using a 0.002 oz Au/bank cubic yard (bcy) cutoff to describe the mineralized volume of the pay gravel. GRE reviewed the wire frame solid and used it to constrain the resource estimate and to prevent excessive dilution from material outside the pay gravel.

Because the project has been previously mined and partially backfilled, current ground topography does not represent the mined-out areas of the project. Ms. Lane created a surface representing the mined-out bottom from a series of topography contour files conducted at various times throughout the mining.

The assay data was composited to a length of 10 feet. Ms. Lane analyzed the composite values and determined that capping was not necessary.

Block grades were estimated by a single pass estimator using the inverse distance cubed (ID3) method. Remaining mineral resources are classified as Measured, Indicated, or Inferred based on the minimum distance to a composite value as follows:

- Measured: 0-75 feet
- Indicated: 76-150 feet
- Inferred: 151-400 feet

The pit-constrained Mineral Resource Estimate for the Chandalar Deposit is presented in Table 1-1.

Table 1-1: Chandalar Mine Pit-Constrained Resources

Classification	Resource Volume (1000s bcy)	Raw Gold Grade (t.oz./bcy)	Raw Gold (1000s t. oz)	Fine Gold (1000s t. oz)
Measured	2,609	0.0302	79	69
Indicated	2,188	0.0265	58	50
Measured & Indicated	4,797	0.0285	137	119
Inferred	771	0.0245	19	16

1) The effective date of the Mineral Resource is May 31, 2021.
2) The Qualified Person for the estimate is Terre Lane of GRE.
3) Mineral Resources are inclusive of Mineral Reserves; Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
4) Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.
5) The Mineral Resource is constrained by a 0.002 raw troy ounce per bank cubic yard grade shell and a 0.004 raw troy ounce per bank cubic yard cutoff (840 fineness) at an assumed gold price of 1,600 $/tr oz, assumed mining cost of 4.50 $/bcy, assumed processing and administrative cost of 7.25 $/bcy, an assumed gold purity of 84%, and pit slopes of 45 degrees. These costs are preliminary estimates (prior to economic analysis).

1.12 Mining Methods

The deposit is a frozen (permafrost) placer that requires conventional open pit mining methods using drill, blast, load, and haul mining methods to mine the frozen gravels, rather than more conventional placer mining methods. Previous operators used the conventional open pit mining methods.

Ms. Lane generated pit shells using Whittle Lerchs-Grossman software using reasonable mining assumptions of $4.50/bcy mining cost, $7.25/ore bcy processing and G&A cost, fineness of 84%, and pit slopes of 45 degrees. Twenty-eight pit shells were generated using gold prices ranging from $300/Au raw ounce to $3,000/Au raw ounce, in increments of $100/Au raw ounce. After a preliminary review of the pit shell data economics, Ms. Lane focused on the $1,400/Au raw ounce pit shell. The pit shell was imported into Geovia GEMS software, where it was segregated into six separate pits or pit pushbacks designated as Lower Pit A, Lower Pit B, Lower Pit C, Upper Pit A, Upper Pit B, and Upper Pit C. Each pit/pushback was designed with haul roads.

In a Multi-Scenario Economic Model, GRE evaluated 30 cases:

- Three phasing options:

Phasing Sequence Designation	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Phase 6
1	Lower Pit A	Upper Pit A	Upper Pit B	Lower Pit B	Lower Pit C	Upper Pit C
2	Upper Pit A	Upper Pit B	Upper Pit C	Lower Pit A	Lower Pit B	Lower Pit C
3	Upper Pit A	Lower Pit A	Lower Pit B	Lower Pit C	Upper Pit B	Upper Pit C

- Five cutover grades (the grade that distinguishes high-grade material from low-grade material): 0.004 opy, 0.006 opy, 0.008 opy, 0.010 opy, and 0.012 opy
- Two production rates:
    - The “ramp-up” production option: 4,000 bcy/day for the first year and 8,000 bcy/day for all subsequent years (a single wash plant can process 4,000 bcy/day)

- 4,000 bcy/day for the project duration

For cutover grades higher than 0.004 opy, which is considered the economic cutoff grade (see equation below), low-grade material would be stockpiled for processing following completion of all high-grade material processing.

To calculate the economic cutoff grade, GRE used typical placer mining costs of \$3.25/bcy processing + \$3.00/bcy G&A.

$$\text{Economic Cutoff Grade: } \frac{\left(\frac{\$7.25}{bcy}\text{ process and G\&A costs}\right)}{(0.84\text{ fineness})\times(\$1{,}600\text{ gold price})} = 0.005\text{ opy, which GRE rounded down to 0.004}$$

The economic cutoff grade is the marginal cutoff grade, i.e., processing and G&A costs are applied, but mining costs are not applied. The costs used in this calculation are preliminary estimates (prior to economic analysis).

Based on the economic analysis of all 30 cases, GRE selected phasing option 2 at a cutover grade of 0.012 opy, and the ramp-up production option, from a rate of 4,000 bcy/day year one, to 8,000 bcy/day year two and after.

Mine scheduling included the production rate, a two 12-hour shift per day, seven-day per week schedule of 120 days between May 17 and September 13 each year. Ms. Lane included a gradual ramp-up to full production during the first year, as shown in Table 1-2.

Table 1-2: Processing Ramp Up

Year/Month	Percent of Production
Year 1/first month	25%
Year 1/second month	50%
Year 1/third month	75%
Year 1/remaining months	100%

In addition, for the ramp-up production options, GRE included gradual ramp-up from 4,000 bcy/day to 8,000 bcy/day for the first two months of the second year.

High-grade material stockpiled during the first-year ramp-up to full production and low-grade stockpiled material was scheduled to be processed after completion of regularly scheduled high-grade material processing. The schedule resulted in a project life of five years.

A final reclamation plan was developed that provides for partial backfilling of the pits. As much as possible, waste material will be backfilled into the pits concurrent with mining activities.

The mine layout would include a process wash site, high-grade and low-grade ore material stockpiles, interim waste dumps required to store waste that cannot be directly backfilled into the pits, and final waste dumps for material not scheduled to be returned to the pits.

The process material and waste would be drilled and blasted using a rotary crawl driller and ammonium nitrate fuel oil (ANFO). Processable material would be hauled using dump trucks from the pit to the wash plant, while waste rock would be hauled using dump trucks to the interim waste dumps or previously mined pit.

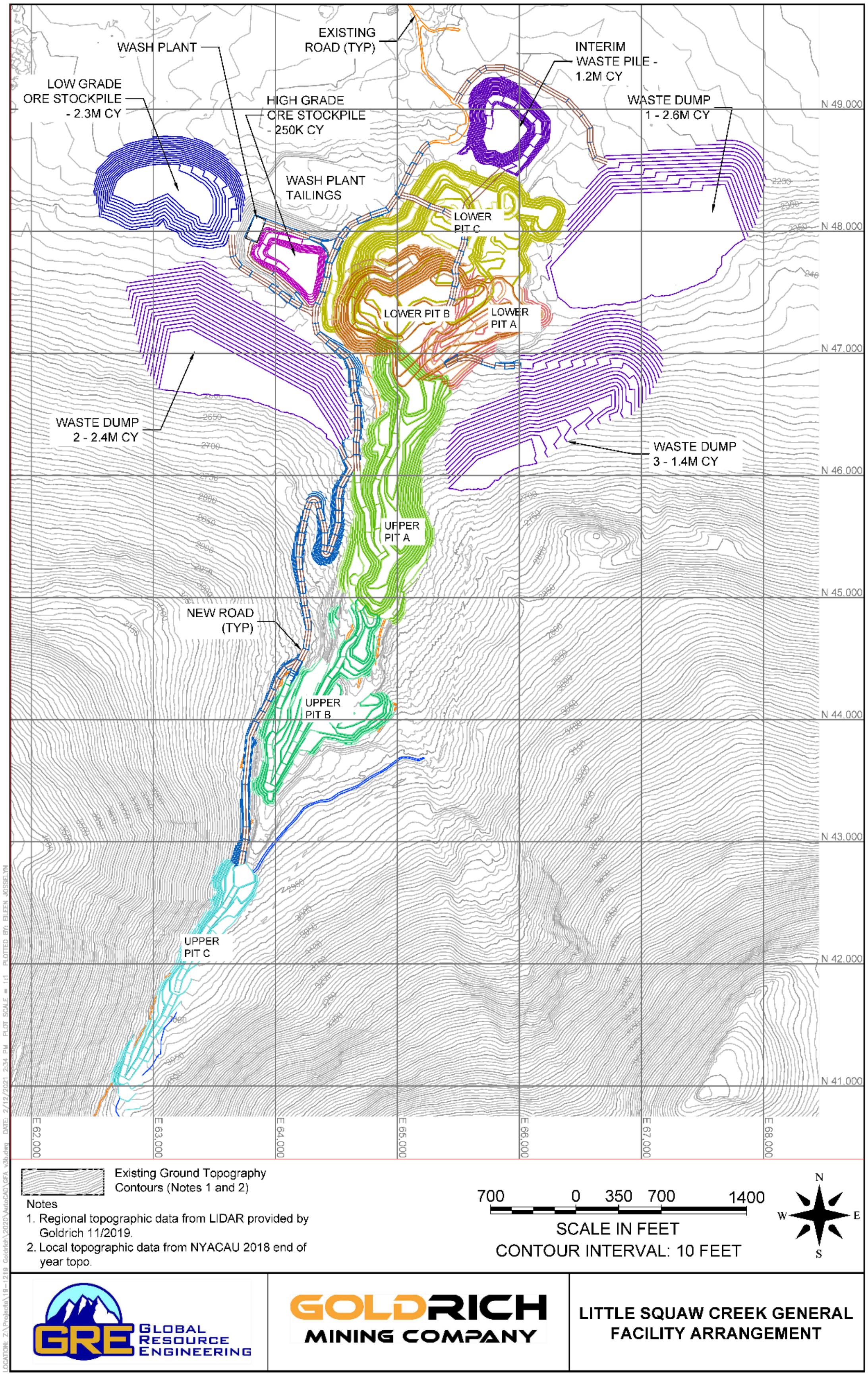
The evaluation uses Caterpillar 745G trucks, with a heaped capacity of 32.7 loose cubic yards, and Caterpillar 982M loaders, with a bucket capacity of 5 loose cubic yards.

Surface and in-pit haul roads are designed with a width of 45 feet. The maximum road grade in pit is 10%.

Ms. Lane used a 20-foot bench height with a 68.2-degree batter angle for the Chandalar pit designs. The 20-foot bench height was selected to satisfy the degree of selectivity anticipated for the grade control effort and to minimize dilution. Catch benches would occur on every bench with a width of 12 feet. This results in overall inter-ramp pit slopes of 45 degrees.

A conceptual General Facility Arrangement is shown in Figure 1-2.

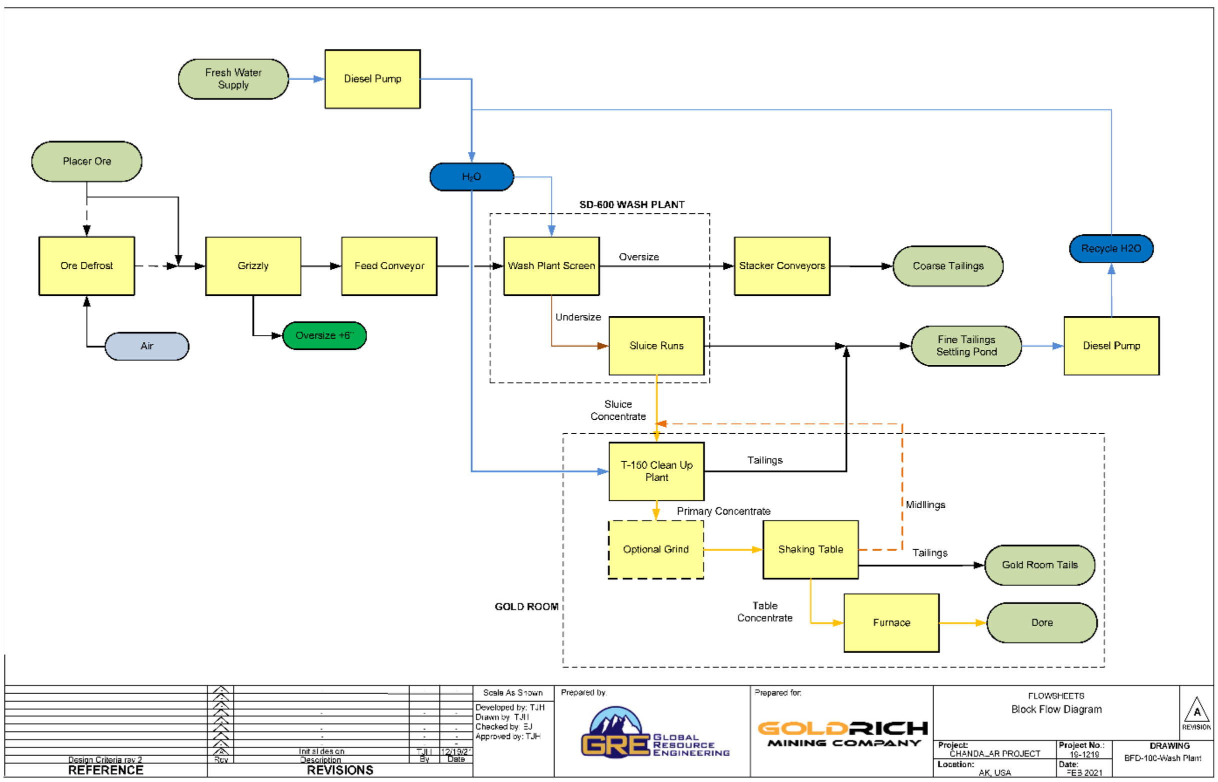
Figure 1-2: General Facility Arrangement



1.13 Recovery Methods

Dr. Harvey has recommended the use of two mobile wash plants shaker systems combined with a feed hopper with a side dumping grizzly, a belt feeder, and plant feed conveyor. The plant consists of a feed receiver with water sprays, a vibrating screen deck, and a double sluice box. The coarse tailings are transported via a series of portable conveyors to a slewing stacker. Having a feed hopper with a grizzly reduces oversize damage to the screen and hopper damage from the mobile equipment. The use of portable conveyors and a stacking conveyor reduce the rehandling of coarse tailings and allow strategic placement in a single handle, reducing rehandle and closure costs. A small gold cleanup system has also been provided that includes a trommel, sluice, and table to produce final doré for sale. Figure 1-3 shows the conceptual process flow sheet for the project.

Figure 1-3: Chandalar Process Flow Diagram



1.14 Project Infrastructure

The project has been successfully explored and has supported mining and washing plant operations since 2012. The current infrastructure includes:

- 27.5-mile (44-km) network of all-season mine roads
- 5,000-foot airstrip capable of landing C-130s (“Hercs”) plus a 4,400-foot abandoned airstrip
- Camp facilities (some owned by NyacAU)
- Generators for the camp and shops
- Shops

- Existing haul roads, pits, waste rock piles,
- Water supply for camp
- Water supply for washing plant
- Gold panning equipment and building

1.15 Market Studies and Contracts

Placer gold is readily marketable and market studies are not needed.

1.16 Environmental Studies, Permitting, and Social or Community Impact

The Chandalar mine has been permitted by the ACOE for reclamation work required to return the property to pre-mining conditions after completion of mining activity.

Permit number POA-2009-366, now on its third modification, was issued on April 8, 2019, and was modified to require the permittee to conduct reclamation on the Upper and Lower Mine Pits, discharging approximately 2,700,000 cubic yards of overburden material into mine pits that were excavated in wetlands and uplands, re-contouring the landscape, and restoring approximately 6,070 linear feet of stream channel. The time limit for completing the work authorized ends on April 30, 2024; however, request for a time extension may be submitted up to one month before permit expiration.

1.17 Reclamation Plan

Mr. Breckenridge reviewed the prior reclamation plan and found that it required significant modification to match the requirements of the future project. As a result, Mr. Breckenridge prepared a revised reclamation plan in accordance with what are believed to be acceptable governmental environmental requirements. The current reclamation plan includes the reclamation to modified streambed topography, and the formation of a post-mining pond. Mine waste piles, stream channels, roads, plant areas, the airstrip, and other impacted ground will be reclaimed and revegetated in accordance with existing laws. Concurrent reclamation will be prioritized in the mining operation.

1.18 Costs and Economics

Capital costs for the project include mining production and support equipment leases that assume 25% down payment of the purchase price and a lease term varying from 20 to 26 quarters, depending on the piece of equipment and when it is needed on the project at 5% interest, a heavy equipment shop and fuel station, process equipment, camp, and site development. Facilities such as dry, cap magazine, guard house, office, and warehouse are either already available on site or will be outfitted from shipping containers. The estimated capital costs total $25.6 million, with initial capital of $15.1 million. GRE obtained vendor quotes for much of the capital costs. Other costs were estimated based on GRE's experience, GNP historical cost data for the project, and InfoMine resources (InfoMine, 2018; InfoMine, 2020).

Operating cash costs are based on a surface mine plan, haul cycle analysis, drill and blast cost analysis, with delivery to the remote mining site by either air (landing strip at the mine site) or by Cat train. Remote labor rates and burdens were used that are consistent with other remote mining operations in the arctic region of Alaska. Power costs for the camp and wash plants is based on generated power using diesel fuel.

The estimated operating costs total $95.2 million, with a cash operating cost of $646/fine Au oz and an all-in sustaining cost of $799/fine Au oz.

Ms. Lane prepared an economic evaluation of the project using a 24-month trailing average gold price of $1,650/Au oz, as stipulated in 17 Code of Federal Regulations (CFR) §229, "*Standard Instructions for Filing Forms Under Securities Act of 1933, Securities Exchange Act of 1934 and Energy Policy and Conservation Act of 1975 – Regulation S-K*," subsection 1300, "*Disclosure by Registrants Engaged in Mining Operations,*" and assuming a raw gold recovery of 100% and ratio of fine gold to raw gold of 84%. The 100% recovery from the wash plants is based on the use of recoverable gold assays for the feed material; recovery losses are accounted for in the feed grade calculations as is typical of placer mining. Smelter charges of 3.6% were applied. Ms. Lane included an order of magnitude, generalized tax calculation that included five-year straight-line depreciation, depletion allowance calculated as 15% of revenues up to a maximum of 50% of before-tax income minus depreciation, Federal tax at 21% applied to the taxable income, and Alaska corporate tax, mining license tax, and production royalty applied to the taxable income.

Note: Ms. Lane is not an expert in taxes and relied on information provided by Goldrich and obtained from on-line searches of U.S. and Alaska tax codes to generate a tax model for the project. The calculations are based on the tax regime as of the date of this 2022 Revised and Amended IA. The tax calculations should be considered approximations because actual tax estimates involve complex calculations that can be accurately determined only during operations.

Sensitivity analyses indicate extreme sensitivity to gold price and moderate sensitivity to both capital and operating costs. A positive valuation, however, was maintained across a wide range of sensitivities on key assumptions. Key economic results with a summarized gold price sensitivity analysis are shown in Table 1-3.

Table 1-3: Key Economic Results

Parameter	Base Case $1,650 Gold	Gold Price Sensitivity Analysis $1,500	$2,000	$2,500
State Royalties:	3%			
Undiscounted Pre-Tax Net Cash Flow:	$75 million	$57 million	$116 million	$175 million
Pre-tax NPV@5%:	$67 million	$51 million	$104 million	$158 million
Pre-tax NPV@7%:	$64 million	$49 million	$100 million	$151 million
Pre-tax NPV@9%:	$61 million	$47 million	$963 million	$146 million
Pre-tax IRR:	256.0%	187.8%	482.7%	1243.4%
After-tax NPV@5%:	$64 million	$50 million	$92 million	$129 million
After-tax NPV@7%:	$61 million	$48 million	$88 million	$123 million
After-tax NPV@9%:	$58 million	$45 million	$84 million	$118 million
After-tax IRR:	138.8%	112.4%	195.5%	275.0%
Undiscounted After-tax Net Cash Flow:	$72 million	$57 million	$103 million	$145 million
After-tax Payback Period:	1.33	1.44	1.19	1.08
All-in Sustaining Costs:	$799/Au ounce			
All-in Costs:	$1064/Au ounce			
Total Operating Costs:	$646/Au ounce			

The project economics shown in this IA are favorable, providing positive NPV at varying gold prices, capital costs, and operating costs.

1.19 Interpretations and Conclusions

The Chandalar district is located in a remote portion of Alaska in the Brooks Range. The district has hosted both placer and lode gold production since the early 1900s. The remote location and harsh winters have limited access, making large scale exploration and mine production difficult. This moderated the success of past efforts. The more recent access to the site via winter road and modern air strips have opened up the area to modern exploration and mining production methods. The Chandalar project hosts both placer and lode gold mineralization; the full potential of each have not been fully tested.

Sensitivity analyses indicate extreme sensitivity to gold price and moderate sensitivity to both capital and operating costs. A positive valuation, however, was maintained across a wide range of sensitivities on key assumptions.

1.19.1 Risks

This IA is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under 17 CFR §229.1300 – (Item 1300). Readers are advised that there is no certainty that the results projected in this initial assessment will be realized. There is inherent engineering and metallurgical risk in all projects at this early stage of development, which are reduced as metallurgical test work and engineering studies, including the prefeasibility and feasibility studies, progress. This project, as with others, would be negatively impacted should gold recovery prove to be less than that used in the study (due to clay or other factors) or positively impacted higher grades are encountered. Similarly, if further geotechnical or reclamation requirements change, the project could be impacted positively or negatively. The largest likely impact would be future changes in the price of gold.

The site has low environmental and permitting risk; however, prior reclamation plans had nearly-complete backfilling. There is a small probability that similar backfilling would be required. This would result in higher closure costs.

The final topographic surface GRE used for the resource estimate stitched together the final 2018 survey completed by GNP on 21 September 2018, which is limited to the areas immediately surrounding the mined areas, and the translated 2019 LIDAR survey. This stitched together surface constitutes a small risk to the estimated resources, but because the 2018 surface would be lower than the 2019 surface due to backfilling that may have occurred in 2019 and due to the fact that there was no mining conducted in 2019, GRE believes the estimated remaining in-situ resource would not change with the 2019 topographic surface but that there could be additional waste backfilled above the estimated resource.

Similarly, the stitched together mining extents of mining could represent a risk if the surfaces provided to GRE do not fully encompass the mining completed by GNP.

1.19.2 Opportunities

Ms. Lane has identified the following opportunities:

- Lode exploration potential
- Other placer deposits on Goldrich controlled ground
- Additional placer gold potential downstream of the currently explored area
- Variable cutoff grade strategies
- Winter pre-stripping of waste
- Mine plan optimization

1.20 Recommendations

The Chandalar placer deposit has been successfully mined, and approximately 156,000 raw ounces of gold remain and appear to be recoverable with conventional mining and placer gold wash plant operation. In order to begin operations, permits need to be acquired, either through negotiation with NyacAU, or new operating permits must be obtained.

A detailed mining plan should be created to carefully plan material movement to concurrently reclaim the mine with operations.

A geologic model of lithology and clays would be helpful to optimize the mine plan.

Ms. Lane recommends the company proceed to a Pre-Feasibility Study (PFS) concurrent with obtaining new operating permits. Once the operating permits are in place and the PFS has been published, Ms. Lane recommends the company proceed with plans to restart operations.

The mine will require modified permits with the ACOE. These should commence as soon as possible to avoid delays.

Ms. Lane recommends compiling all existing geologic data, property-wide, and conducting new mapping and geophysical surveys of high priority targets in preparation for drill campaigns.

The coordinate system for the project is a local Imperial coordinate system; some surveying has been conducted in this local coordinate system while others have been performed in UTM NAD83. Ms. Lane recommends adding the UTM coordinates to the drill hole database and using that grid and metric measurements for all project work going forward.

2.0 Introduction

This Technical Report was revised and amended on February 24, 2023 from the original Report issued on May 5, 2021 and the Revised and Amended Report issued on November 10, 2022.

Goldrich Mining Company (Goldrich) is a U.S. based mineral resource company focused on developing the Chandalar gold district in Alaska, USA. Goldrich has retained Global Resource Engineering, Ltd. (GRE) to prepare this Initial Assessment (IA) for the Chandalar deposit within the greater Chandalar Gold Project area, located along the southern flank of the Brooks Range in north-central Alaska.

2.1 Purpose of the Technical Report

This report presents the results of the IA based on data collected from 2007 through 2017, including a mineral resource estimate updated to incorporate drilling completed in 2013 and 2017, as well as mining completed from 2009 through 2018. This report is intended to fulfill 17 Code of Federal Regulations (CFR) §229, "*Standard Instructions for Filing Forms Under Securities Act of 1933, Securities Exchange Act of 1934 and Energy Policy and Conservation Act of 1975 – Regulation S-K*," subsection 1300, "*Disclosure by Registrants Engaged in Mining Operations*." The mineral resource estimate presented herein is classified according to 17 CFR §229.1300 – (Item 1300) Definitions.

This IA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under 17 CFR §229.1300 – (Item 1300). Readers are advised that there is no certainty that the results projected in this IA will be realized.

2.2 Qualified Persons and Personal Inspection

The Qualified Persons (QPs) responsible for this report are:

- Hamid Samari, PhD, MMSA 01519QP, Senior Geologist, GRE
- J. Todd Harvey, PhD, SME 4144120RM, Director of Process Engineering, GRE
- Terre A. Lane, MMSA 01407QP, SME 4053005RM, Principal Mining Engineer, GRE
- Richard Hughes, PE (AK EM-5531), Goldrich
- J. Larry Breckenridge, PE, (CO 38048) Principal Environmental Engineer, GRE

Ms. Lane and Dr. Samari conducted a personal inspection of the Chandalar deposit on September 4 and 5, 2019. Dr. Samari, Dr Harvey, Ms. Lane, and Mr. Breckenridge are collectively referred to as the authors of this IA.

Table 2-1 identifies QP responsibility for each section of this report.

Table 2-1 List of Contributing Authors

Section	Section Name	Qualified Person
1.1 – 1.3	Summary	Terre Lane
1.4 – 1.9	Summary	Hamid Samari
1.10	Summary	J. Todd Harvey
1.11 – 1.12	Summary	Terre Lane
1.13	Summary	J. Todd Harvey
1.14 – 1.15	Summary	Terre Lane
1.16 – 1.17	Summary	Larry Breckenridge
1.18 – 1.20	Summary	Terre Lane
2	Introduction	Terre Lane
3	Reliance on Other Experts	Terre Lane
4	Property Description and Location	Terre Lane and Larry Breckenridge
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Terre Lane
6	History	Terre Lane
7	Geological Setting and Mineralization	Hamid Samari
8	Deposit Types	Hamid Samari
9	Exploration	Hamid Samari
10	Drilling	Hamid Samari
11	Sample Preparation, Analyses and Security	Hamid Samari
12	Data Verification	Hamid Samari
13	Mineral Processing and Metallurgical Testing	J. Todd Harvey
14	Mineral Resource Estimates	Terre Lane
15	Mineral Reserve Estimates	Terre Lane
16	Mining Methods	Terre Lane
17	Recovery Methods	J. Todd Harvey
18	Project Infrastructure	Terre Lane
19	Market Studies and Contracts	Terre Lane
20	Environmental Studies, Permitting and Social or Community Impact	Larry Breckenridge
21	Capital and Operating Costs	Terre Lane
22	Economic Analysis	Terre Lane
23	Adjacent Properties	Terre Lane
24	Other Relevant Data and Information	Terre Lane
25	Interpretation and Conclusions	Terre Lane
26	Recommendations	Terre Lane
27	References	Terre Lane

Note: Where multiple authors are cited, refer to author certificate for specific responsibilities.

2.3 Sources of Information

The following information was provided by Goldrich:

- Drill hole records
- Project history
- Sampling protocols

- Geological and mineralogical data and interpretations
- Data, reports, and opinions from third-party entities
- Prior permitting documents and reclamation plans
- Gold assays from original records and reports
- Claim information and land position
- Royalty agreements

A portion of the background information and technical data presented in this report was obtained from a previous Technical Report on the Chandalar Gold Project (Barker, et al., 2009), and from a recent, unpublished technical report specific to the Chandalar deposit (Mendham, et al., 2018). Historical documents and data sources used during the preparation of this report are cited in the text, as appropriate, and are summarized in current report Section 27.

2.4 Units

Unless otherwise stated, all measurements are reported in U.S. imperial units, volumes are presented as bank cubic yards, and grade is reported as troy ounces per bank cubic yard (opy).

3.0 Reliance on Other Experts

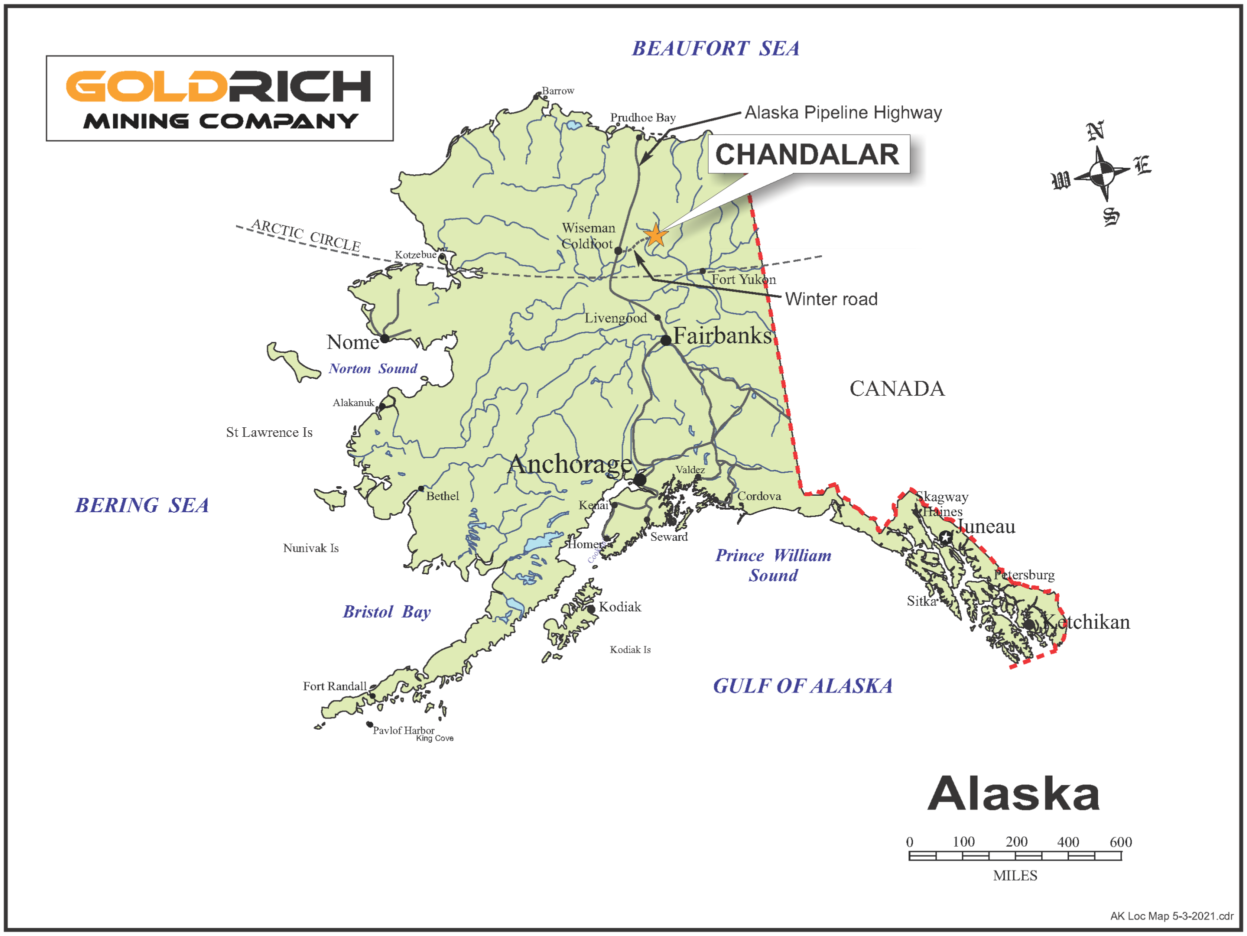
GRE QPs have fully relied upon and disclaims responsibility for information provided by Goldrich regarding property ownership, mineral tenure, and permitting and environmental aspects of the project. Such information is presented in Sections 4 and 5 of this report.

4.0 Property Description and Location

4.1 Location

The Chandalar Gold Project is located in north-central Alaska, north of the Arctic circle, on Alaska state land surface along the southern flank of the Brooks Range. The property is situated at approximately 67°34'N latitude and 148°10’W longitude (Figure 4-1), roughly 190 air miles (306 kilometers [km]) north of Fairbanks and 48 air miles (77 km) east-northeast of the community of Coldfoot.

Figure 4-1: Location of Chandalar Mining District, Alaska



Source: https://www.sec.gov/Archives/edgar/data/59860/000105291812000312/grmcs1jun2812.htm

Coldfoot is the community nearest to the Chandalar district and offers basic municipal amenities such as lodging, food, fuel, and communications, as well as a state-maintained airport. Coldfoot is an important service center along the Dalton Highway, which parallels the Trans Alaska Pipeline and provides the only road access to the Prudhoe Bay oil fields on Alaska’s North Slope.

4.2 Property Description

Goldrich (formerly the Little Squaw Gold Mining Company [LSGMC]) holds approximately 23,000 acres (9,300 hectares [ha]) of minerals rights covering much of the Chandalar Mining District. The property consists of 23 patented federal mining claims and 197 unpatented state mining claims (Figure 4-2). The claim block covers all or portions of:

- Sections 1, 12, 13 and 24 of R4W, T31N
- Sections 1-11, 15-22, 27, 28, 34 and 35 of R3W, T31N
- Sections 4-6 of R2W, T31N
- Sections 22-28 and 31-36 of R3W, T32N
- Sections 28-33 of R2W, T32N

Map coverage of the project area is provided by the Chandalar C-3 15-minute topographic quadrangle. Approximate boundaries of the claim block are as presented in Table 4-1 as Universal Trans Mercator (UTM) coordinates using North American Datum (NAD) 27.

Figure 4-2: Chandalar Gold Project – Goldrich Mining Claims, Significant Historical Mines, Access Roads

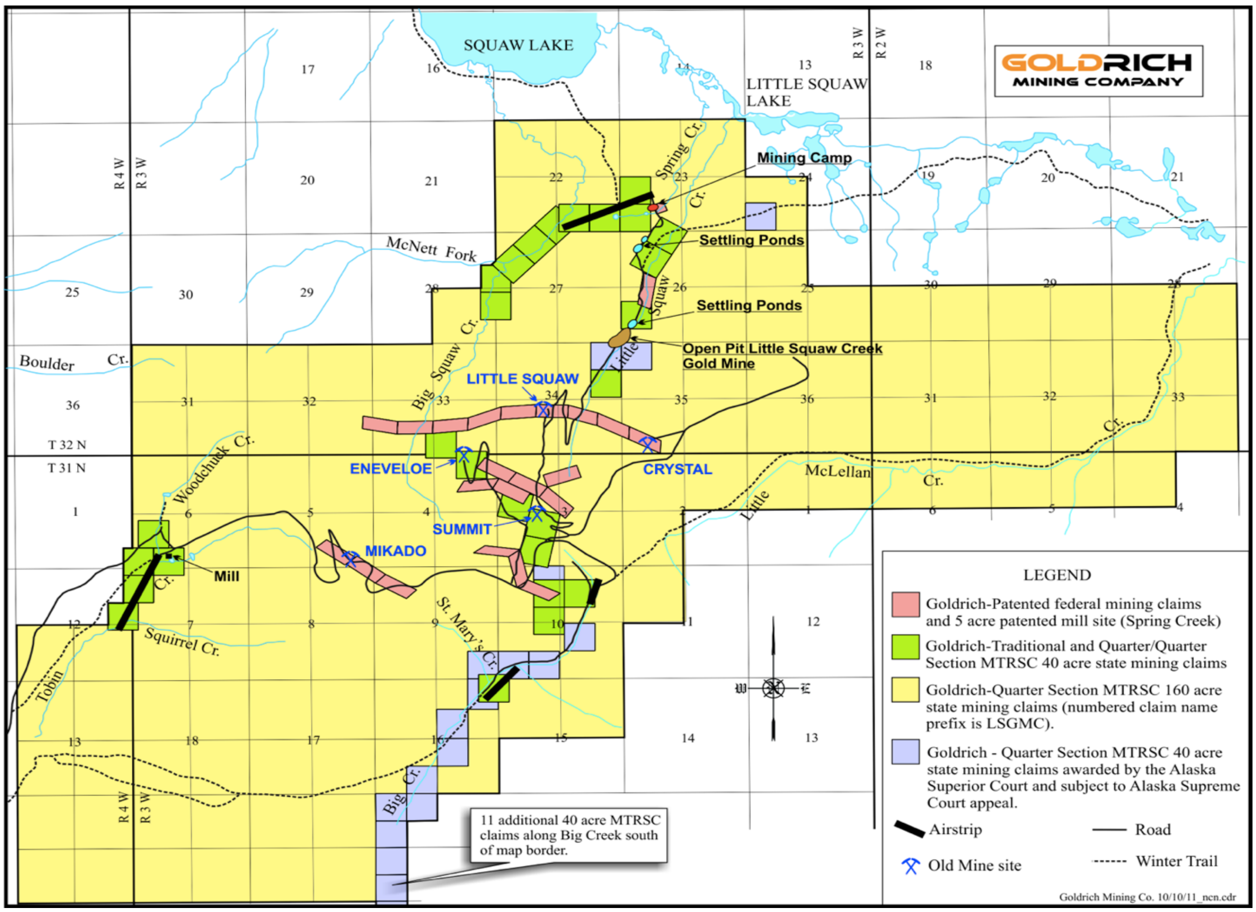


Table 4-1: Approximate Claim Block Boundaries

Boundary	Lat-Long	UTM NAD 27
North boundary	67° 35’ 28” N	7,497,644 meters N
South boundary	67° 29’ 24” N	7,486,406 meters N
West boundary	148° 21’ 11” W	442,448 meters E
East boundary	147° 58’ 52” W	458,298 meters E

The 23 patented claims are comprised of 22 Federal mineral claims plus one mill site claim, together totaling 426.5 acres (172.6 ha), clustered in four separate tracts that cover most of the known historical lode gold system. The unpatented claims include 59 Alaska state claims of 40 acres (16.2 ha) each and 138 Alaska state claims of 160 acres (64.8 ha) each.

The 197 state claims, identified by state ADL/Serial numbers, are listed in Table 4-2.

Table 4-2: State Claim List

State ADL/Serial #s	Number of Claims	Comments
40-acre state claims		
319523 - 319533	11	
515440- 515445	6	
515447	1	
515452	1	
515468 - 515474	7	purchased 2003
641349 - 641368	20	court award
645852	1	
649584 - 649592	9	purchased 2010
663873-663874	2	
667284	1	
total 40-acre claims	**59**	
160-acre state claims		
553169	1	
641504 – 641558	55	
645239 – 645246	8	
653068 – 653091	24	
657650 – 657652	3	
661131 – 661132	2	
663875 - 663886	12	
709124 – 709156	33	
Total 160-acre claims	**138**	

Twenty-nine of the 40-acre claims were located prior to September 2003; the remaining 30 of the 40-acre claims together with all 138 of the 160-acre claims were located between September 2003 and 2011 under new Alaska state Meridian-Township-Range-Section-Claim (MTRSC) staking regulations. These newer claims were purposely staked to overlap parts of the older claims to cover any fractional land segments. For this reason, the total area of the unpatented claims is not exactly known but is thought to be approximately 22,858 acres (9,250 ha).

The 23 patented Federal claims are identified by U.S. Patent number in Table 4-3.

Table 4-3: Federal Claims

Claim Group	No. of Claims	U.S. Patent No.	Acres	Dates
Little Squaw lode	5	1022769	96.215	1929
	1	1036360	20.102	1930
	1	1036361	19.582	1930
	2	1088433	41.101	1937
Bonanza Gold lode	3	1036358	61.446	1930
	2	1085903	37.209	1936
	1	1085904	20.655	1936

Claim Group	No. of Claims	U.S. Patent No.	Acres	Dates
Mikado lode	3	1024558	58.964	1929
Star lode	3	1036359	50.553	1930
Squaw Creek placer	1	1036362	15.718	1930
Spring Creek millsite	1	1091946	4.961	1938

4.3 Mineral Title, Taxes, Royalties, and Fees

The State of Alaska recognizes mining claim locations according to a state-wide township-range-section grid, with the claim location positions filed with the state having legal priority over locations staked in the field. The Chandalar state mining claims are recorded at the Fairbanks Recording District and at the Alaska Division of Mining, Land, and Water in the Alaska Department of Natural Resources, Fairbanks, Alaska. The patented claims are located relative to two mineral location monuments established at the time of the mineral surveys; these documents are on file with the U.S. Bureau of Land Management office in Fairbanks.

Alaska state mining claims are maintained by payment of annual rentals and the performance of assessment work. Annual rental costs are based on the number of years of continuous activity since a mining claim was first located, a sliding scale for the first 10 years ranging from $40 to $205 for each 40-acre claim and from $165 to $825 for each 160-acre claim. The state offers a 50% claim rental rebate after the first year of location as an incentive to stake 160-acre MTSRC claims; this rental rebate does not apply to Goldrich's Chandalar Mine, all of which are more than one year old. Annual assessment work expenditures are $100 for each 40-acre claim and $400 for each 160-acre claim. Work expenditures exceeding the required annual amount are bankable for up to four years. The total costs to maintain Goldrich's state mining claims for 2020 at Chandalar are US $187,845 consisting of US $125,945 for claim rentals and US $61,100 for assessment work. Exploration expenditures by the company from 2006 to 2020 applicable against future annual assessment work requirements total US $20.1 million.

Alaska's tax and regulatory policy is widely viewed by the mining industry as offering favorable environment for establishing new mines in the United States. The mining taxation regimes in Alaska have been stable for many years. There is regular discussion of taxation issues in the legislatures, but no changes have been proposed that would significantly alter their current state mining taxation structures. The economics of any potential mining operation on Goldrich's properties would be particularly sensitive to changes in the State of Alaska's tax regimes. Amendments to current laws, regulations, and permits governing Goldrich's operations and the general activities of mining and exploration companies, or more stringent implementation thereof, could cause unanticipated increases in exploration expenses, capital expenditures, or future production costs, or could result in abandonment or delays in establishing operations at Goldrich's Chandalar property. Although management has no reason to believe that new mining taxation laws that could adversely impact the Chandalar property will materialize, such an event could happen in the future.

At present, Alaska has a 7% net profits mining license tax on all mineral production (AS 43.65), a 3% net profits royalty on minerals from state lands (AS 38.05.212) (where Goldrich holds unpatented state mining claims), and a graduated annual mining claim rental beginning at $1.03/acre. Alaska state corporate income tax is 9.4% if net profit is more than a set threshold amount. Alaska has an exploration incentive credit program (AS 27.30.010) whereby up to $20 million in approved accrued exploration credits can be deducted from the state mining license tax, the state corporate income tax, and the state mining royalty. All qualified new mining operations are exempt from the mining license tax for 3 1/2 years after production begins. The Chandalar mine does not qualify for this exemption.

4.4 Litigation

In April 2012, Goldrich Placer, LLC ("GP"), a subsidiary of Goldrich, and NyacAU, LLC ("NyacAU"), an Alaskan private company, entered into a joint venture agreement (the "Operating Agreement") and formed a 50:50 joint venture company, Goldrich NyacAU Placer LLC ("GNP"), to bring Goldrich's Chandalar placer gold properties into production with NyacAU acting as managing partner.

According to the terms of the Operating Agreement, NyacAU was to fund GNP with a line of credit (LOC1) to bring the placer gold properties into commercial production as defined in the agreement. Under a separate Security Agreement between GNP and NyacAU, NyacAU was entitled to record a security interest in all placer gold production from the placer claims as collateral for repayment of fifty percent (50%) of LOC1.

In 2017, Goldrich, its subsidiary and the joint venture, as claimants, filed an arbitration statement of claim before a three-member Arbitration Panel ("the Panel"), against NyacAU and its affiliates; NyacAU, LLC ("NyacAU"), BEAR Leasing, LLC, and Dr. J. Michael James, as respondents. In 2018, the respondents filed a counter-claim against Goldrich, its subsidiaries and certain members of the Company's current and former management, the counterclaim respondents. The arbitration claim alleged, amongst other things, claims concerning related-party transactions, accounting issues including capital vs. operating leases, interpretation of the joint venture operating agreement, allocation of tax losses between the joint venture partners, and unpaid amounts due Goldrich relating to the Chandalar Mine.

The arbitration is still in process. Some of the on-going matters and Panel's rulings that affect the mine are as follows:

- As of December 31, 2018, GNP had not achieved commercial production as required under the Operating Agreement. GNP was deemed by the Panel to have been dissolved during 2018 and, as of December 31, 2020, GNP was in the process of liquidation. The Panel ruled that NyacAU was to continue as manager of the liquidation. Except for equipment needed for reclamation, most of the heavy equipment and the wash plant were removed in March through mid-April 2019.

- Goldrich owns all its claims, but NyacAU is the holder of the mine permits. However, GNP no longer has the right to mine, and NyacAU, as holder of the mine permits, only has the obligation and liability of reclamation. NyacAU began reclamation of the mine in 2019 and is responsible for future reclamation costs. NyacAU has indicated they will not transfer the permit without also transferring the reclamation obligation, of which they believe to be approximately $3 million. Goldrich has indicated they will not accept transfer of the permit together with the reclamation obligation, which Goldrich believes to be substantially greater, unless the reclamation obligation is reconciled with any obligation to pay LOC1. Both parties are in discussion to attempt to reach an agreement for the transfer of both the permit and the reclamation obligation, no transfer of either, or some other arrangement.

- The Panel calculated a tentative balance of LOC1 at $16,483,271 as of June 2019. This balance will be adjusted for any additional awards and/or adjustments made by the Panel. The Panel ruled that LOC1 cannot be increased for costs incurred after mining operations have ceased, including costs for reclamation. Mining operations ceased on September 21, 2018. This deprives NyacAU of a security interest in 50% of future placer gold production at the site to repay NyacAU for expenses incurred subsequent to the cessation of mining operations. Concerning LOC1, the agreements between GNP and NyacAU are silent concerning what happens if GNP is dissolved and is no longer producing gold, the basis of calculation, timing of remittance and other key factors related to repayment if mining activities were to be undertaken again.

Additional details concerning the arbitration and the rulings by the Panel are contained in Goldrich's form 10-K and exhibits for the year ended December 31, 2019 and Goldrich's forms 10-Q for the first, second, and third quarters of 2020 that have been filed with the U.S. Securities and Exchange Commission (SEC).

Related to the arbitration and GNP, in June 2015, Goldrich raised net proceeds of $1.1 million through the sale of 12.5% of the cash flows of GP, that would be received in the future from its interest in GNP, to Chandalar Gold, LLC ("CGL") and GVC Capital, LLC, ("GVC"), both of which are non-related entities. Goldrich retained its ownership of its 50% interest in GNP but, after the transaction, subject to the terms of the GNP Operating Agreement, GP would have effectively received approximately 44%, CGL would have effectively received 6% (12% of Goldrich's 50% of GNP = 6%), and GVC would have effectively received 0.25% (0.5% of Goldrich's 50% of GNP = 0.25%) of any distributions produced by GNP. During the nine months ended September 30, 2020, Goldrich purchased 595,000 membership units, or approximately 49% of CGL's membership units, for $25,000. This provides Goldrich with 49% of any distributions produced by GNP and paid to CGL; however, Goldrich and CGL have not decided how distributions are to be made now that GNP is dissolved.

4.5 Permits and Water Rights

Goldrich maintains a State of Alaska water right, issued in 1985, allowing the withdrawal of up to 3,000 gallons (11,360 liters) of water per minute for placer mining, and 72,000 gallons (272,550 liters) per day for lode mining. The water can be withdrawn from any of the local streams specified in the permit for use from April through October. The water right is maintained by paying an annual $50 administrative fee and by demonstrating some beneficial use of the water at least once in any 5-year period.

4.6 Environmental Liabilities

The Chandalar project includes several sites of historical mining activity, mainly placer gold operations along local creek valleys. Most of the placer material was exploited by underground drift mines driven in frozen ground at depth, and as a result there are few non-reclaimed placer tailings remaining at surface, and no outstanding reclamation is required. Tailings impoundment ponds from a permitted lode mining and milling operation in the late 1980s were sealed in the early 1990s with approval of the Alaska Department of Environmental Conservation (ADEC).

Several minor issues related to historical lode gold mining and processing activities, specifically at the Tobin Creek mill site, were identified in the 1980s by the ADEC. In 1993, the ADEC approved plans to have the property lessee at the time reprocess about 200 cubic yards (153 cubic meters) of fill contaminated with low levels of mercury in a small area adjoining the old Tobin mill and assay lab. The lessee declared bankruptcy soon thereafter and did not perform the clean-up; Goldrich carries a current $100,000 accrued remediation cost to execute the approved plan.

Mr. Breckenridge completed a first principal estimate for reclamation of the Chandalar mining area disturbed by GNP's mining activities to the ACOE permit-approved reclamation plan, which included backfilling all waste into the existing mining pits and re-establishing pre-mining ground contours. The estimate showed reclamation costs of approximately $18.0 million. GRE has proposed a new reclamation plan that will include concurrent backfilling of the new pits to new contours that would include ponds and would not bring the pits to pre-mining contours. In addition, some waste material would remain on waste piles. The costs and work to complete the new reclamation plan are included in this IA and the results reported are net of all costs for reclamation.

5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Access

Air access to the project area is available year-round via aircraft from Fairbanks to Goldrich's 4,400-foot (1,220-meter) airstrip at Squaw Lake. Overland access is available only during the winter season and is provided by a 55-mile-long (88.5-km-long) trail from Coldfoot to the state-owned airport at Chandalar Lake. From Chandalar Lake, the project is accessed either by a 17-mile (27-km) winter dozer trail to Goldrich's airstrip and camp at Squaw Lake or by a 7-mile (11-km) pioneer road along Tobin Creek to the Tobin Creek mill and mine camp. The Squaw Lake and Tobin Creek camps are connected to all major prospects in the district via a 27.5-mile (44-km) network of all-season mine roads.

During the winter months, heavy equipment and bulk supplies for use in the project are hauled by truck from Fairbanks over the Dalton Highway to Coldfoot, and then by Cat train over the winter trail to Goldrich's Squaw Lake camp (Figure 4-1). The winter road season usually extends from mid-January through late-March. Goldrich obtains an annual permit from ADNR to secure use of the overland route.

In January 2007, the State of Alaska obtained a right-of-way access into the Chandalar area via the Coldfoot-Chandalar Lake winter trail. This right-of-way gives the state sole management authority over the access route and allows the construction of a permanent all-season road along the trail. All-season road access into the Chandalar area would improve year-round access to the Chandalar Gold Project and positively impact exploration and development activities throughout the Chandalar district.

5.2 Climate and Physiography

The Chandalar district covers a portion of the south flank of the Brooks Range. The area is bounded to the west by Chandalar Lake and the North Fork of the Chandalar River, and to the east by the valley of the Middle Fork of the Chandalar River. Both rivers drain south out of the Brooks Range. Elevations range from 2,000 feet (610 meters) in the Squaw Lake lowland, the site of Goldrich's exploration camp, to more than 5,000 feet (1,520 meters) in the adjacent mountain peaks.

Local terrain is rugged and steep, consisting of talus-covered hill slopes and deeply incised, alluvium-filled valleys. The higher elevations are barren of vegetation except for moss, lichen, and some grasses, and the lower country is mantled by relatively continuous spruce forests. Bedrock exposures are largely limited to ridge crests and creek bottoms. The ubiquitous talus slope cover is a product of repeated freezing and thawing, a process known as solifluction, which causes a down-slope creep of soil and rock in the form of rock lobes and rock glaciers. Permafrost in the area is continuous and extends to depths of several hundred feet.

Snowmelt generally occurs toward the end of May and is followed by an intensive 90-day growing season with more than 20 hours of daylight and daytime temperatures ranging from 60 to 80°F (15.6 to 26.7°C). Freezing temperatures return in late August, with freeze-up typically occurring by early- to mid-October. Winter temperatures, particularly in the lower elevations, can drop to -50°F (-45.6°C) or colder for extended periods. Annual precipitation is 15 to 20 inches (38 to 51 centimeters), occurring mostly as late summer rain and early winter snowfall.

5.3 Infrastructure

Since 2004, Goldrich has made several significant improvements to project infrastructure, including construction of a 4,400-foot (1.3 km) airstrip, which was later replaced with a 5,000-foot (1.5 km), and adjoining 25-person camp at Squaw Lake and repair and upgrade of most of the existing 27.5-mile (44-km) network of mine roads. The mine roads provide all-season access to all major gold prospects in the district area and to older camps and airstrips at the Tobin Creek mill site and Big Creek placer site. The Squaw Lake airstrip can readily handle multi-engine cargo aircraft up to C-130 size. The adjoining camp consists of various equipment repair facilities, storage units, office buildings, and sleeping and kitchen facilities. The camps at Tobin Creek and Big Creek contain several serviceable structures for use as temporary exploration camps.

As there is no electrical grid in northern Alaska, the project relies on diesel generators to power operations.

Water is sourced from a natural spring located on a patented 5-acre (2-ha) mill site claim at the Squaw Lake camp and airstrip site. The spring flows 140 gallons (530 liters) per minute (gpm) at a year around temperature of 40°F (4.4°C). Water for the wash plant and sluicing operation is from Big Squaw Creek.

The land holdings and surface rights held by Goldrich are sufficient for all planned operations and cover most potential placer or lode exploration targets.

6.0 History

6.1 Ownership History

William Sulzer, a former Governor of New York and owner of Chandalar Mines Company, became the major lode claim owner in the Chandalar district in 1909, effectively establishing the modern day Chandalar project. In 1926, he merged with Chandalar Gold Mines, Inc., and in 1937, the project was acquired by Chandalar Gold Mines, Ltd., of Toronto. Sulzer continued to maintain a significant interest in the project and helped to finance exploration and development throughout the district until his death in 1942.

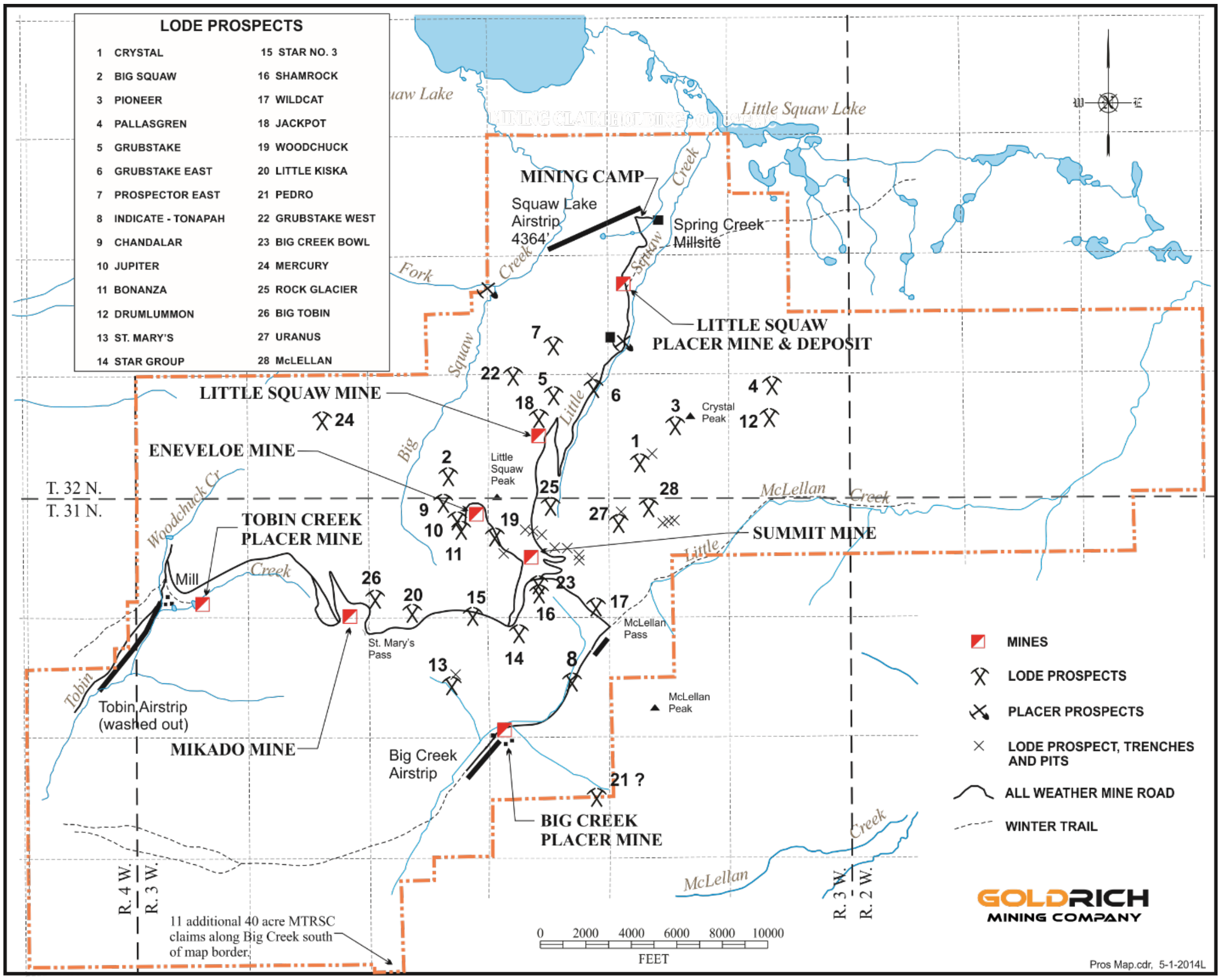
In 1946, Eskil Anderson acquired the Chandalar interests of the Sulzer estate, and, in 1959, incorporated the holdings as Little Squaw Mining Company. In 1968, the company was renamed Little Squaw Gold Mining Company (LSGMC), and in 1972 merged with Chandalar Gold Mines, Ltd., effectively giving LSGMC control of most of the Chandalar project. In 2003, Anderson sold his interests in LSGMC and the Chandalar project to LSGMC and other outside interests. In 2008, the company was renamed Goldrich Mining Company, as it remains today. In 2012, Goldrich and NyacAU formed Goldrich NyacAU Placer, LLC (GNP), a 50/50% joint-venture company, managed by NyacAU, to mine the various placer deposits located within the Chandalar project area.

6.2 Placer Gold Exploration History

The long and storied history of prospecting and mining in the Chandalar district is described in detail in several recent reports: Wolff (1997); Barker and Bundtzen (2004); Strandberg (1990); Barker et.al. (2009). Rich gold placer deposits were first discovered in the district in 1905 along Little Squaw Creek, within the present day Chandalar project area. The initial discovery of placer gold was followed by a flurry of prospecting activity, ultimately leading to the discovery of bedrock lode deposits. Maddren (1910) reports that by 1909, four principal auriferous quartz veins had been identified. To date, more than 30 lode deposits and historic lode prospects are known to occur within the Chandalar project area alone.

Placer gold discoveries in the Chandalar district were relatively deep under frozen overburden. By 1916, shallow, open-cut placer gold mines were playing out and attention shifted to developing placer drift mines (underground operations). Most notable was the Little Squaw Bench, including the Mello Bench, where about 30,000 troy ounces (oz) of gold (Au) were reportedly recovered from gravel averaging 0.96 oz Au/cubic yard (Strandberg, Jr., 1990). The Mello Bench is located on the lower portion of Little Squaw Creek in the southwest quarter of Section 26, T32N, R3W of the Fairbanks Meridian. Elsewhere within the Chandalar District, by 1916, gold placers were similarly developed on Big Creek and St. Mary's Creek (see Figure 6-1). Drift mining continued through the 1920s but declined in the 1930s as the remaining ground was deeper or lower grade, or in many cases not frozen. In 1933, a rich pay streak on Tobin Creek was discovered. Mechanized mining was not introduced to the Chandalar District until after World War II, when Chandalar Mining Company began mining Big Creek.

Figure 6-1: Regional Mine Prospect Map



Mining and exploration activity have been carried out in the Chandalar project area sporadically since 1967, when the Tobin Creek placer mine was expanded by the Chandalar Gold Mining and Milling Company. A 100 ton per day (tpd) mill was constructed in 1969, and further exploration, including drilling on both lode and placer targets, was completed throughout the property by a variety of lease holders through 2003, when Goldrich (as LSGMC) acquired the project.

Early expeditions into the Chandalar region by the U.S. Geological Survey specific to the Chandalar district included Mertie (1925), who mapped much of the Chandalar Quadrangle and visited many of the mines. The U.S. Geological Survey published geologic maps of the Chandalar Quadrangle (Brosgé, et al., 1964; Brosgé, et al., 1970), which presented stratigraphy and structural interpretations and provided the first modern isotopic age dates of plutonic and metamorphic rocks in the southern Brooks Range. In 1970, the Alaska Division of Mines and Minerals provided the first detailed geologic map of the Chandalar district at 1:40,000 scale (Chipp, 1970). Fluid inclusion studies by Ashworth (1983; 1984) were followed by the Rose et al. (1988) study of the mineralizing fluids responsible for the gold deposition in the Chandalar.

6.3 Historical Gold Production

6.3.1 District-Wide Production

Estimates of historical production given by Strandberg (1990), as updated by Barker and Bundtzen (2004) and Walters (2020), include documented lode gold production of 8,223 ounces, primarily attributed to the Mikado (6,699 ounces) and Summit (1,462 ounces) mines. Records exist only for lode production from 1971 through 1987, so the actual total District-wide production is likely larger. The Alaska State Department of Natural Resources estimates that the district has produced 17,400 ounces of gold from lodes (Szumigala, et al., 2005).

The documented District-wide placer gold production is 104,913 ounces, including 44,209 ounces of production by Goldrich and NyacAU from the Chandalar open-pit placer operation from 2009 to 2018. Approximately 60% of the placer production has been from Chandalar, 26% from Big Creek, with the remaining 14% from Tobin Creek. The District's actual historical placer production could be considerably larger as the quantity of gold taken from early open-cut hand mining in the upper drainage areas of these creeks (some of very high grade) is only partially known.

6.3.2 Chandalar Mine Production

In 2009, Goldrich completed an alluvial gold mining test on Chandalar Mine. The pilot program involved a mining test that extracted approximately 594 "raw" ounces of placer gold, equivalent to about 488 ounces of fine gold. The test mining yielded valuable geologic, mining, and engineering data that encouraged Goldrich to ramp-up the project into extraction in the spring of 2010. The major findings of the test indicated that the mineralized material is a continuous but variably mineralized horizon. There are specific horizons within it that are up to 20 feet thick containing the richest gold grades. The mineralized material is about forty percent composed of gravel, cobbles, and boulders set in a sixty percent matrix of fine silt. It is not frozen below twelve to fifteen feet of depth but is nicely compacted and stands well when opened. Because of the high silt content, the mineralized material and overburden expands by over forty percent in volume when it is mined and converted into loose cubic yards.

During the summer of 2010, Goldrich started a small mining operation at the Chandalar Mine, the site of their previous test mining operation, known as the Chandalar Mine. Full realization of the intended project was inhibited by a shortage of working capital. By the end of the 2010 mining season, Goldrich had extracted 1,906 ounces of gold concentrate, from which approximately 1,522 ounces of fine gold and 259 ounces of fine silver were extracted.

Since 2009, the Chandalar Mine has produced approximately 44,209 ounces of fine gold, as summarized in Table 6-1.

Table 6-1: 2009 to 2018 Gold Production at Chandalar Mine

Year	Approximate Ounces of Placer Gold	Ounces of Fine Gold
2009	594	488
2010	1924	1509
2011	0	0
2012	0	0

Year	Approximate Ounces of Placer Gold	Ounces of Fine Gold
2013	936	694
2014	0	0
2015	4,697	3,852
2016	10,203	8,227
2017	14,676	12,339
2018	20,357	17,100
Total	53,387	44,209
2009	594	488
2010	1924	1509
2011	0	0

6.4 Historical Mineral Resource Estimates

Estimates of lode gold resources for unmined portions of the Mikado, Little Squaw, Summit, and Eneveloe lodes were compiled by Strandberg (1990). He based the estimates on polygonal blocks measured from longitudinal sections constructed from all available historical surface and underground mapping, sampling, drill data and other information collected by many previous operators in the district. He provides almost no details regarding the specific parameters used in the estimations other than a tonnage factor of 12 cubic feet/ton (equivalent to a specific gravity of 2.86).

Cautionary statement: The mineral resource estimates presented in the following table pre-date 17 Code of Federal Regulations (CFR) §229, "*Standard Instructions for Filing Forms Under Securities Act of 1933, Securities Exchange Act of 1934 and Energy Policy and Conservation Act of 1975 – Regulation S-K*," subsection 1300, "*Disclosure by Registrants Engaged in Mining Operations*.". These historical results describe mineralization relevant to ongoing exploration but have not been verified by the authors of this current report and should not be relied upon. They are reported here for historical purposes only.

The historical in-place lode gold resource estimates compiled by Strandberg (1990), a Registered Professional Engineer (# 1613-E), are summarized in Table 6-2.

Table 6-2: Historical In-Place Lode Gold Resource Estimates Compiled by Strandberg (1990)

Mine	Short Tons	Au oz/ton	Au total oz	Classification
Mikado	110	1.10	121	Measured
	5,200	1.05	5,439	Indicated
Little Squaw	1,986	1.55	3,084	Indicated
Summit	1,375	3.52	4,835	Indicated
	3,615	2.13	7,695	Inferred
Eneveloe	5,360	1.00	5,360	Inferred
Total	**8,671**	**1.55**	**13,479**	**Measured + Indicated**
	8,975	**1.45**	**13,055**	**Inferred**

Strandberg (1990) additionally tabulated conceptual estimates of "Lode Exploration Targets" for these areas totaling 630,000 ounces of gold at grades like those in his resource table. These conceptual estimates appear to be based on various questionable assumptions and are reported here for historical purposes only.

Following Goldrich's 2007 to 2009 exploration along Little Squaw Creek, a mineral resource estimate was prepared by Paul L. Martin, P.E., specific to unmined Chandalar Mine placer gold resources (Barker, et al., 2009). Martin's mineral resource classifications are considered consistent with 17 CFR §229.1300 – (Item 1300) reporting standards (although for clarity are not defined or reported as Item 1300 resources).

The 2009 mineral resource estimate for the Chandalar deposit was calculated by standard sectional resource-polygon methods and tested for accuracy using semi-variograms for grade and grade multiplied by thickness. The mineralized fluvial section within the proposed resource area was estimated to average 82 feet thick, and the overburden was estimated to average 50 feet thick along a pay channel strike distance of 5,129 feet. The resource calculation method used is a common resource estimate technique that was used successfully at the Valdez Creek Operations (located south of Fairbanks, Alaska) to determine resources, mine plans, and reserves. The cross-sectional method is described in detail in the SME Mining Engineering Handbook, Volume 1, pages 352 and 353 (SME, 2011) and in the Open Pit Mine Planning and Design, Volume 1, Fundamentals, pages 176 to 196 (Hulstulid, et al., 2006). A cross section was prepared for each drill fence line and on section, the pay and overburden areas were determined based on the drill hole assays. Within the pay zone, pay polygons were drawn to define the limits of influence for a drill hole and were usually drawn one-half the distance between holes. To determine the appropriate search criteria and number of holes for measured, indicated, and inferred resources, a detailed Data Analysis (geostatistical evaluation) was prepared (Martin, et al., 2010).

For the purpose of estimating placer resources, the gold-bearing section, including two to three feet of bedrock of each drill hole (or the maximum depth of 210 feet), was composited and treated as a bulk deposit. This gave two mine sections: 1) the barren overburden section and 2) the potentially economic gold-bearing or pay section. Profiles for each drill line were created, and the top-of-pay and base-of-pay was drawn using a nominal fine gold cut-off grade of 0.004 ounces per cubic yard (opy). Barren and sub-economic intervals within the pay section were included to derive the average grade for the drill hole, giving the bulk amount of gold potentially available for extraction. Right and left limits to the resource blocks were selected where a clear truncation of the pay streak was observed on a drill fence due to diminished values of gold or a geologic feature such as steeply rising bedrock, otherwise they were inferred 50 feet beyond the end of a line that was open to further continuity of mineralization.

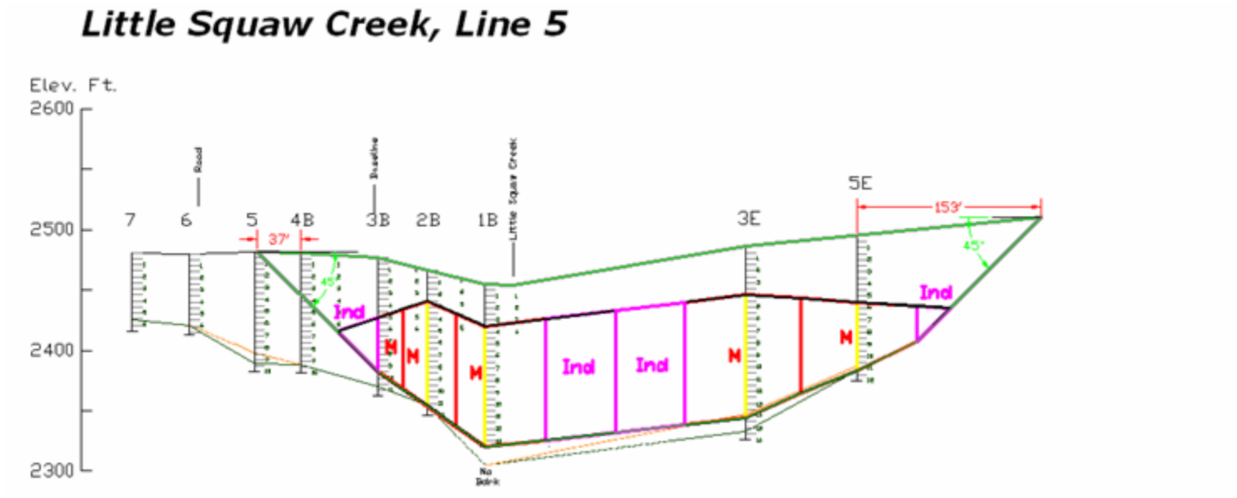
To determine the corresponding overburden volume and stripping ratio for each fence line, a 45-degree highwall was drawn to the surface on the lateral extents of each section. The interception point of the highwall to the surface was then plotted on plan view, and a crest or pit limit was drawn on plan view.

A typical cross section of a drill hole fence line with polygons between drill holes is shown in Figure 6-2.

The experimental variography analysis presented in Table 6-3 was used to establish a basis for resource influences for measured, indicated, and inferred resource classification and to demonstrate continuity of pay section grade and thickness of the deposit. A reduced percentage of the calculated variography ranges for measured and indicated was used.

The 2009 mineral resource estimate, as completed by Paul Martin, for measured and indicated volumes in bank cubic yards (bcy) and fine gold ounces per bcy for mineralized gravel is summarized in Table 6-4.

Figure 6-2: Typical Cross-Section Showing Drill Hole Fence Line with Placer Resource Polygons



Section	Measured Pay BCY	Indicated Pay BCY	Inferred Pay BCY	Total Pay BCY	Meas Grade Pay Au Oz/BCY	Ind Grade Pay Au Oz/BCY	Inf Grade Pay Au Oz/BCY	Total Grade Pay Au Oz/BCY	Measured Au Fine Ozs	Indicated Au Fine Ozs	Inferred Au Fine Ozs	Total Au Fine Ozs	Total Overburden BCY	Total Material BCY	Strip Ratio Ob:Pay	Section Influence Ft	Au Oz Linear Ft
5	99,079	634,315	362,959	1,096,353	0.0170	0.0175	0.0174	0.0174	1,688	11,105	6,331	19,124	728,594	1,824,947	0.66	747.46	25.58

M=Measured
Ind=Indicated

Drill holes on Little Squaw Creek

Line #	Hole #	Total Depth (feet)	Bedrock Depth (feet)	Overburden Thickness (feet)	Pay Gravel Thickness (feet)	Ore Grade for Pay Section (oz/bcy)
LS-L5	5E	120.0	108	55	55	0.0137
LS-L5	3E	160.0	139	40	102	0.0318
LS-L5	1B	135.0	no bk	35	100	0.0069
LS-L5	2B	120.0	111	25	87	0.0091
LS-L5	3B	115.0	107	50	45	0.0039

Table 6-3: Summary of Semi-Variogram for Range (Ao), Chandalar Deposit

Variogram	Range Major (feet)	Minor (feet)	Percent Major (feet)	Minor (feet)	Resource Range Major (feet)	Minor (feet)	Resource Class
Grade	121.00	121.00	83%	83%	100.00	100.00	Measured
Thickness	361.00	180.50	0%	0%	-	-	
Grade X Thickness	680.00	340.00	74%	74%	500.00	250.00	Indicated
Grade X Thickness	680.00	340.00	100%	100%	680.00	340.00	Inferred

Major axis = 0 degrees = North/South Minor Axis = 90 degrees = East/West

Table 6-4: 2009 Measured and Indicated Resources

Resource Status	Total Pay Gravel (bcy)	Grade Pay Gravel (Au Oz/bcy)	Total Au Fine (Troy Ozs)
Fan			
Measured	1,136,376	0.0243	27,622
Indicated	5,308,654	0.0239	126,857
Measured + Indicated	6,445,030	0.0240	154,479
Inferred	830,750	0.0196	16,271
Canyon			
Measured	453,130	0.0272	12,316
Indicated	2,203,440	0.0247	54,481

Resource Status	Total Pay Gravel (bcy)	Grade Pay Gravel (Au Oz/bcy)	Total Au Fine (Troy Ozs)
Measured + Indicated	2,656,570	0.0251	66,797
Inferred	570,916	0.0365	20,822
Total			
Measured	1,589,506	0.0251	39,938
Indicated	7,512,094	0.0241	181,338
Measured + Indicated	9,101,600	0.0243	221,276
Inferred	1,401,666	0.0265	37,093

7.0 Geological Setting and Mineralization

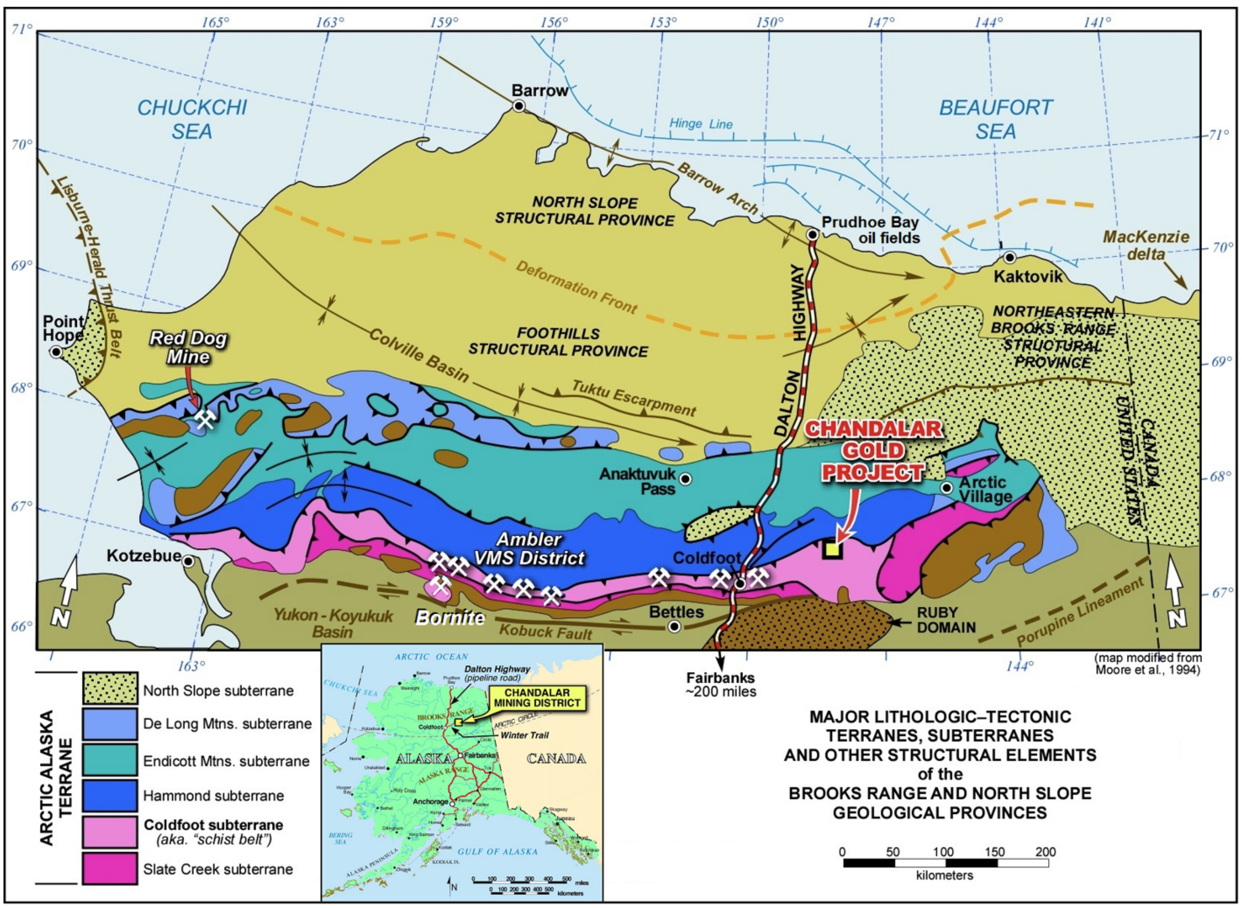
Portions of this report section are adapted from Barker, et al. (2009) and Mendham et al. (2018). The author has reviewed this information in detail and finds it reasonably accurate and suitable for use in this report.

7.1 Regional Geology

7.1.1 Geologic and Tectonic History

Most of Alaska's North Slope, including the Brooks Range, is underlain by a complex assemblage of Late Proterozoic to Cretaceous, composite tectono-stratigraphic bedrock units collectively known as the Arctic Alaska terrane (Figure 7-1).

Figure 7-1: Major Lithologic-Tectonic Terranes, Subterranes and Other Structural Elements of the Brooks Range and North Slope Geological Provinces



Source: Mendham and Lam (2018)

In the southern Brooks Range, the Arctic Alaska terrane is composed of five subterranes, which are (from north to south) the Coldfoot, Hammond, Endicott, Delong Mountains, and North Slope subterranes. Each of the five subterranes is bounded by regional, east-west trending, northerly-directed, low-angle thrust faults. These thrust faults are compressional features formed by the collision of an ancient continental landmass to the north with an assemblage of oceanic rocks (known as the Angayucham Terrane) to the south. The southern outcrop boundary of the Arctic Alaska terrane lies along the Kobuk suture zone, where the continental rocks were overthrust by the southerly oceanic rock assemblage in late-Jurassic time, roughly 160-140 million years (Ma) (Moore, et al., 1994).

The Chandalar district is largely or entirely underlain by the Coldfoot subterrane, which consists mainly of Proterozoic to Lower Paleozoic metasedimentary schist intruded and overlain by bimodal metavolcanics and granitic rocks of Devonian age. The Coldfoot subterrane contains the Ambler sequence, which hosts world class volcanogenic massive sulfide deposits west of the Dalton Highway corridor (Hitzman, et al., 1982). In the Chandalar district, metamorphic rocks of the Coldfoot subterrane include schist, phyllite, and slate, with minor amounts of meta-gabbro and meta-diabase. Based on sparse fossil control found west of Wiseman, Brosgé and Reiser (1964) assign a Devonian age to meta-sedimentary rocks in the Chandalar quadrangle, though without better local constraint a potential age range of Late Proterozoic to Devonian is considered. Based on geologic map coverage to date, all of the gold-quartz deposits in the Chandalar district are hosted in the Coldfoot subterrane, as are gold-quartz vein deposits in the Nolan-Wiseman and Wild Lake areas.

7.1.2 Regional Tectonic Setting

Northeast, east-west, and to a lesser extent, northwest trending structures are the major features on the south flank of the Brooks Range (Dillon, 1989; Moore, et al., 1994; Chipp, 1970; Brosgé, et al., 1964). Large fold structures with 5- to 15-mile wave lengths generally trend northeast across the central Chandalar quadrangle. The Baby Creek batholith, about ten miles west of the Chandalar project, forms the core of a large northeast-trending anticlinorium (Duke, 1975; Dillon, et al., 1996). The structural deformation of the region is typified by several stacked thrust panels that successively overlie a basement of unknown composition and age. According to Dillon (1982; 1989), two or possibly three periods of regional dynamo-thermal metamorphism have affected the layered rocks in the Coldfoot and Hammond subterranes, imprinting S_{1-3} cleavage surfaces. The first prograde metamorphism, which increases in intensity in a southerly direction, resulted in the development of regionally penetrative layer-parallel cleavage (S_1) and development of upper greenschist to lower amphibolite facies metamorphic conditions. Till (1992) and Dusel-Bacon (1994) cite evidence for Proterozoic and possibly Paleozoic pro-grade blueschist and retrograde amphibolite facies metamorphism in the Brooks Range, mainly west of the project area in the Ambler River area. During Jurassic to mid-Cretaceous time (K-Ar ages 154-172 Ma), the entire Brooks Range schist belt was subjected to low-pressure, high-temperature amphibolite facies conditions (Hitzman, et al., 1982; Dillon, 1989; Dusel-Bacon, 1994). During a second prograde metamorphism, the Hammond subterrane was subjected to two periods of progressive deformation producing northward-verging folds, semi-penetrative cleavage (S_{2-3}), and development of the lower greenschist facies retrograde metamorphism. Biotite and muscovite developed in this last period of metamorphism, having cooling ages ranging from 90-120 Ma (Turner, et al., 1979; Dillon, et al., 1989).

The major period of crustal shortening, thrust faulting, and isoclinal folding has been determined by faunal and isotopic control ages to be the Neocomian (130-140 Ma), which coincided with the collision of the Arctic Alaska terrane along the Kobuk suture zone and proto-Pacific “Angayuchum Ocean” (Mull, 1989). A much younger Albian to Turonian (80-110 Ma) uplift and plutonic event post-dated the Neocomian crustal shortening event and resulted in “gravity slide” tectonism that may have taken place during the late phase of regional greenschist facies, retrograde metamorphism (Dillon, 1989). The prominent S_{2-3}

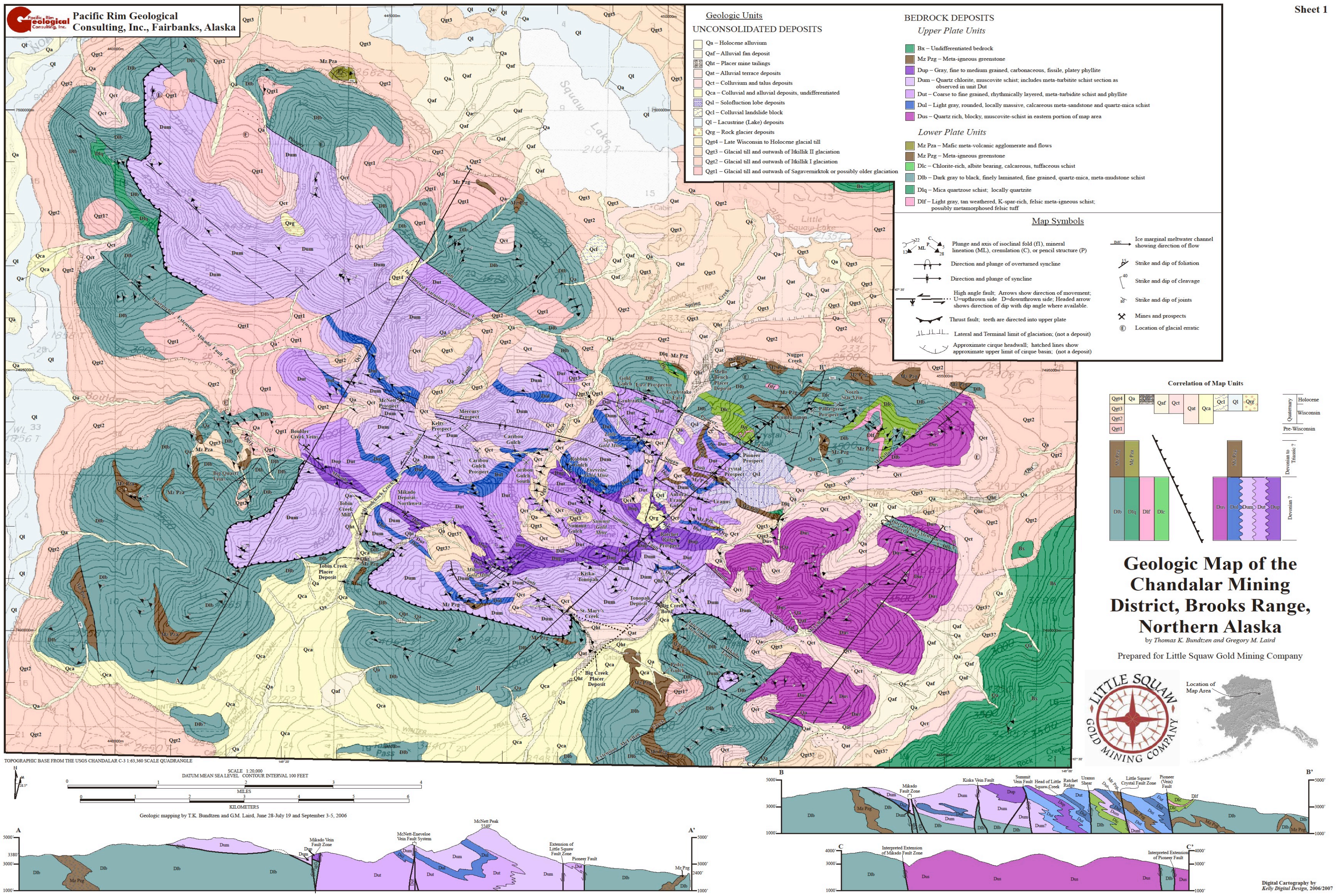
foliation surfaces observed in the phyllite and schist of the Coldfoot subterrane in the Chandalar area were likely developed during one or more of the Cretaceous dynamo-thermal events.

7.2 Local Geology

The Chandalar district occurs near the central part of the 1:250,000 Chandalar topographic quadrangle, the geology of which was first mapped by the U.S. Geological Survey in 1964 (Brosgé, et al., 1964). Detailed, district-wide geology was mapped in 1970 at 1:40,000 scale by the Alaska Division of Mines and Minerals (Chipp, 1970). In 2006, Goldrich commissioned a detailed geologic and structural map of the Chandalar project area itself at 1:20,000 scale (Figure 7-2) (Bundtzen, et al., 2007; Pacific Rim Geological Consulting, Inc., 2007).

Metasedimentary and meta-igneous rocks of the Coldfoot subterrane underlie the Chandalar district, trending roughly N50-60ºW. These rocks have been subjected to several periods of regional metamorphism, ranging from lower greenschist to lower amphibolite facies, most recently during Cretaceous time. Large intrusive centers containing several batholith-sized igneous bodies, mostly also of Devonian age, are located 10 to 12 miles (16 to 19 km) west of the project area. According to published maps and reports, there are no post-metamorphic igneous rocks present in the Chandalar district. The metamorphic rocks in the Chandalar project area are divided into Upper Plate and Lower Plate sequences that are separated by a major low-angle thrust fault plane (Figure 7-2). Most of the known gold occurrences in the district are in rocks of the Upper Plate Sequence.

Figure 7-2: Geologic and Structural Map of the Chandalar District and Project Area, 1:20,000 Scale Map



GLOBAL RESOURCE ENGINEERING

7.2.1 Lower Plate Sequence

Rocks of the Lower Plate crop out across the southern and northern flanks of the property, as shown in green on Figure 7-2 and Figure 7-3. The rocks have all been regionally metamorphosed to upper greenschist facies. The Lower Plate is comprised of several lithologic units consisting of black schist, phyllite, slate, and quartzite. The dominant unit is a quartz-graphite-chlorite black schist that lies in thrust contact with the overlying schist section of the Upper Plate. Other distinctive units are green calcareous tuffaceous schist, quartzite, meta-felsite tuff, and a small unit of agglomerate.

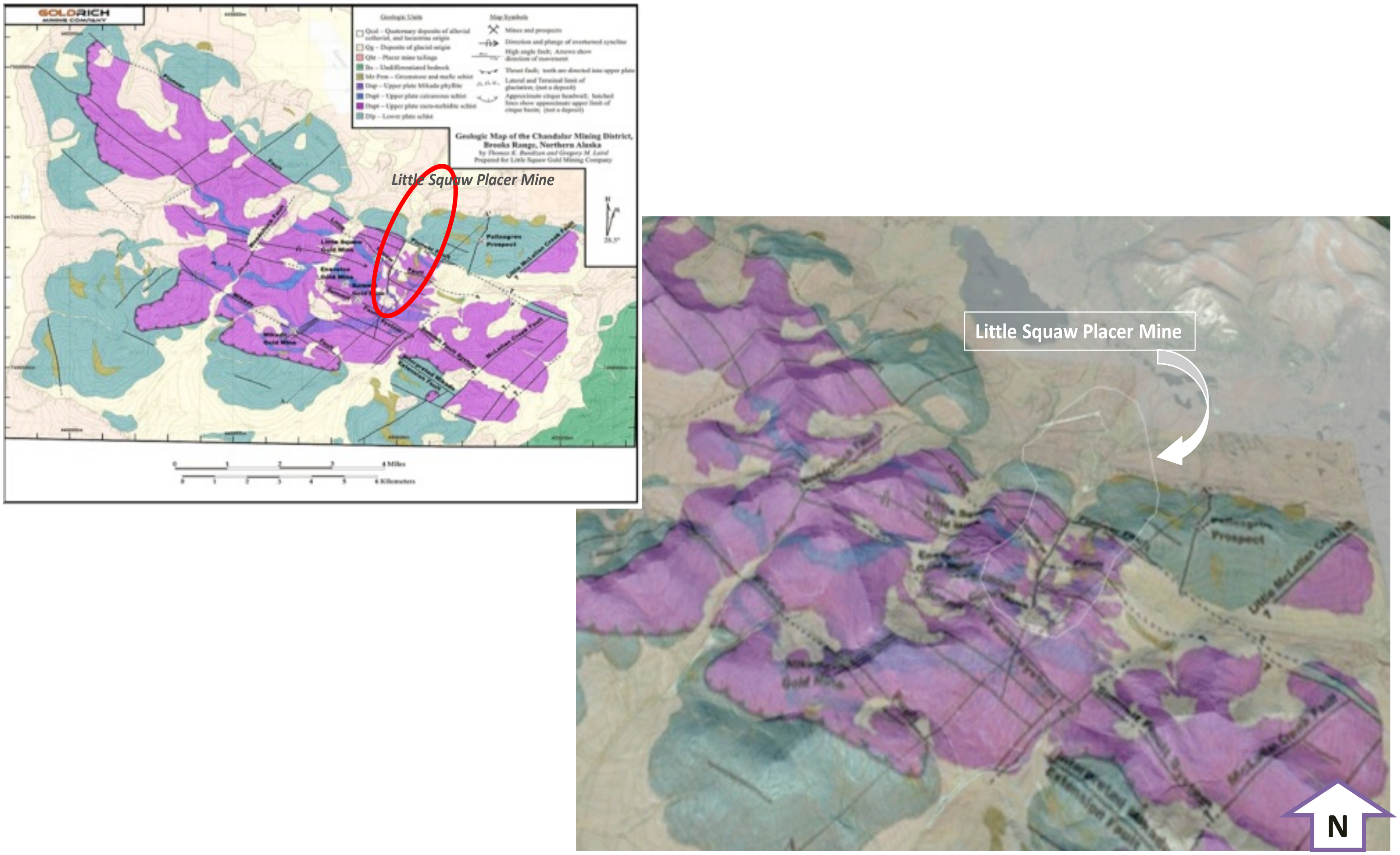
The Lower Plate rocks are locally intruded by weakly foliated to non-foliated greenstone sills or dikes of diorite-gabbro composition and igneous bodies of felsic to intermediate composition. The igneous bodies in the Lower Plate sequence have been assigned a Devonian age (Brosgé, et al., 1964), although age-dates from similar bodies elsewhere in the southern Brooks Range often return much younger metamorphic ages (Turner, et al., 1979).

7.2.2 Upper Plate Sequence

Rocks of the Upper Plate crop out at the higher elevations on the property and are therefore better exposed than rocks of the Lower Plate (Figure 7-3 and Figure 7-4). The Upper Plate sequence has been locally subdivided into five metamorphic units, as follows (Pacific Rim Geological Consulting, Inc., 2007):

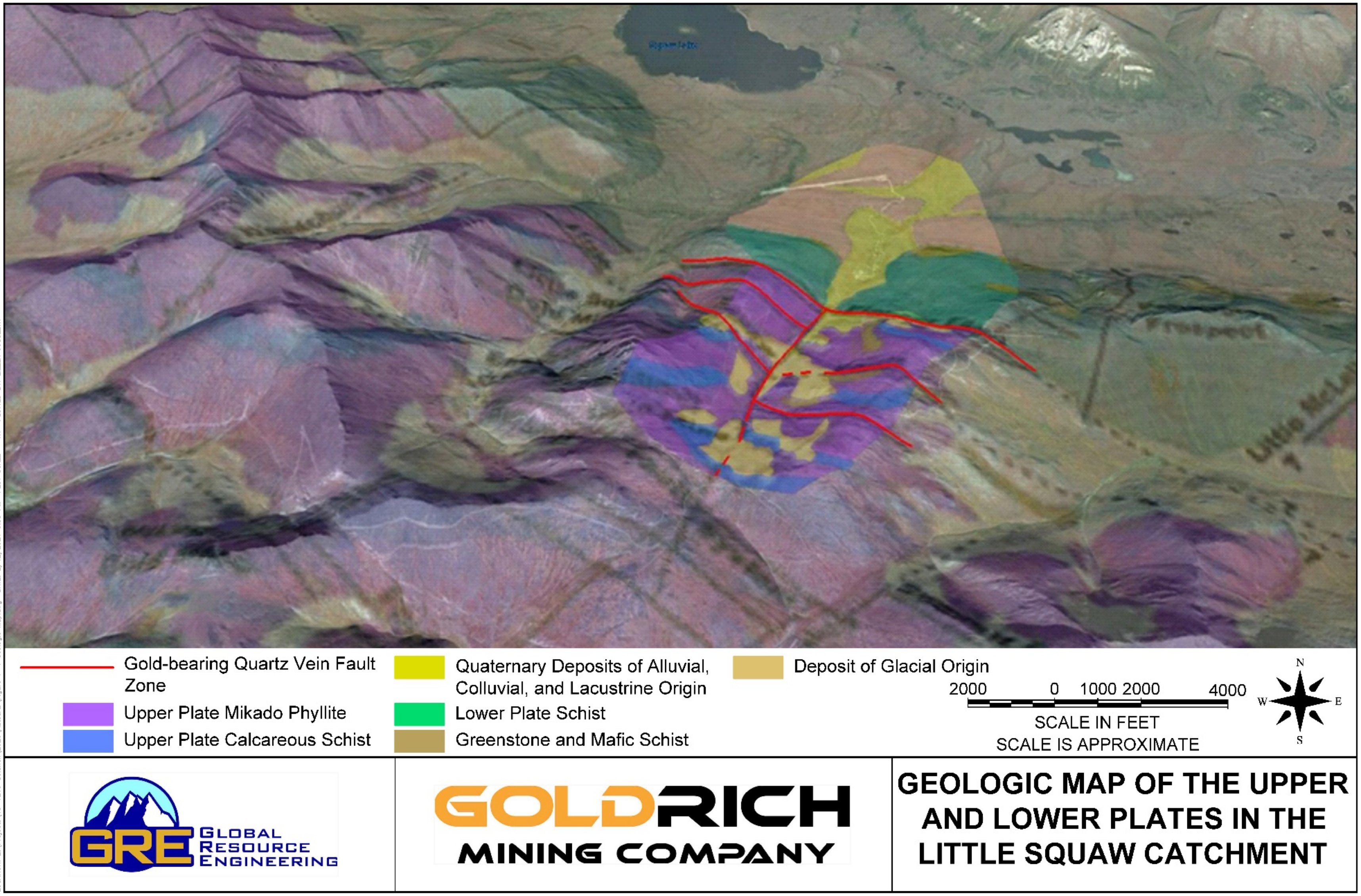
- Mikado phyllite – sheared grey-to-black, highly carbonaceous and micaceous, pyrrhotite-bearing fissile schist and phyllite near the base of the Upper Plate. Weathers readily, rarely crops out, often forms slippery scree slopes. It is the principle constituent of rock glacier and landslide block deposits in the area. May be relatively thin, possibly no more than 300 feet (± 90 meters) thick but is more extensive than mapped and apparently underlies much of the area of the Upper Plate. It possibly represents a chlorite-altered, structurally pulverized lithologic horizon, i.e. a “mylonite.” Closely associated with much of the gold-quartz vein mineralization in the project area and likely source of much of the placer gold.
- Quartz-chlorite-muscovite schist, locally a meta-turbidite schist – resistant to weathering, dominates outcrops and detrital debris in the area. Comprises about 65% of the Upper Plate in the project area.
- Quartzose meta-turbidite schist – distinctive fine- to coarse-grained layered unit, gradational with the quartz-chlorite-muscovite schist (above). Resistant to weathering, often occurs along ridge crests. Comprises about 15% of the Upper Plate in the project area.
- Distinctive light-colored, often green, actinolite-bearing, calcareous, meta-sandstone – locally forms resistant massive fine-grained outcrops. Found in the footwall of several disparate vein systems, e.g., Little Squaw Mine and Mikado Mine.
- Light grey, blocky, quartz-rich, muscovite-oligoclase schist, locally garnet-bearing – of higher metamorphic grade than the other Upper Plate units, found in the eastern part of the project area.

Figure 7-3: Simplified Geologic Map of the Chandalar District



Source of Aerial Photo: Google Earth

Figure 7-4: Geologic Map of the Upper and Lower Plates in the Little Squaw Catchment



Source of Aerial Photo: Google Earth

7.2.3 Local Structural Setting

The dominant trend of local fold axes is east-west to northwest. These folds range from outcrop scale to regional scale and open to overturned isoclinal folds. North-northeast trending folds with open large wavelengths and amplitude occur in both plates due to east-west compression.

There is a pronounced low-angle compositional discordance between the Upper and Lower Plates in the southern portion of the project area, which is illustrated by the occurrence of numerous greenstone sills and dikes and extensive graphitic black schist in the Lower Plate. Age of the décollement surface separating the Upper and Lower Plates is uncertain but may pre-date the orogenic gold-quartz veins if associated with the Neocomian period of crustal shortening and thrust faulting. A K-Ar date of 111 Ma on sericite separated from an auriferous quartz lens from the Mikado Mine is available (Newberry, written communication, 2006) and is similar to reported mineralization dates from the Wiseman area. It remains unclear if the gold-quartz veins cut across the décollement surface or have been displaced by the low-angle fault movement.

A system of west-northwest, northwest, and northeast trending, high-angle and deep-seated faults transect the district and dominate deformation. Known gold-quartz veins generally are subparallel to the west-northwest faults, although several lower grade veins (e.g., Big Tobin) have been found associated with the northeast faults.

Most northwest-trending faults with associated mineralization have had recurrent movement (see Figure 8-1), as best seen on the Summit Fault. Mapping by Chipp (1970) indicates that the Mikado Fault displaces the Mikado Phyllite more than 500 vertical feet on its southwest footwall. A similar structural horst juxtaposes Lower Plate black schist against quartz-rich blocky schist of the Upper Plate in lower McLellan Creek valley.

Textures observed in mineralized veins along fault structures provide some of the best evidence of recurrent movement history along the northwest-striking faults. Multiple laminae of slickensides at the Chandalar mine 100 Level and at the Crystal vein give the vein a ribbon appearance. The Mikado vein-fault system typically consists of highly pulverized pinching and swelling veins/lenses due to recurrent movement along the fault zone. The veins splay, and several subparallel mineralized zones are generally present. The Mikado fault has apparently had a complex history of movement that occurred before, during, and after injection of hydrothermal gold-bearing quartz lodes.

There is evidence of post mineralization movement and shearing on the Summit fault and likely the Kiska vein systems. At the Summit, the mineralization along 4,000 feet of strike length has been off-set by repeated right-lateral northeast faults, yet the well-defined Summit shear zone shows no similar displacement, indicating significant movement since the auriferous veins were emplaced.

7.2.4 Surficial Geology

Prior to the Pleistocene glacial advances, ground surface in the project area was low-relief, ranging in elevation from 5,000- to 5,500 feet. This paleo-surface is evidenced by remnants in small areas near the Summit and Kiska prospects, a broad ridge above the old Mikado Mine, and a series of adjoining sculptured knife ridges that divide the local watersheds. Ancestral drainages, including Big Squaw and Little Squaw Creeks and Little McLellan and Nugget Creeks, were immature second-order streams that formed relatively large alluvial fans on the base level lowland to the north. At the onset of the Pleistocene period, glaciation initially resulted in trunk glaciers that followed the ancestral river valleys of the upper forks of the Chandalar River southward out of the high elevations of the Brooks Range. The glacial advances bifurcated, and lower energy branches of the glaciers encroached on the north flank of the Chandalar district.

Pre-glacial surficial features were buried under ice and, ultimately, lateral moraines and meltwater silt, clay, and glaciofluvial marginal deposits. The pre-glacial fluvial fan on Little Squaw Creek is well defined on drill lines 1.2 to 4. The ancient fan lies immediately north, where the stream exits a buried canyon. There, the sedimentary section can be divided into a barren upper glacial till section and a lower gold-bearing fluvial section. Locally, the contact between the two is sharp, but typically there is a mixed zone between the two and, overall, the contact is gradational. Within the mixed zone, interlayered fluvial gravel and glacial till are interpreted to be thin fan and/or delta deposits laid down as glaciers advanced and retreated in the Lake Creek valley, probably multiple times. As the climate continued to cool, Pleistocene glacial effects on the different drainages varied. Valley glaciers formed cirques above 4,500 feet and scoured fluvial gravel out of some sections of Big Squaw Creek, McNett Fork, Tobin Creek, Woodchuck Creek, Squirrel Creek, and both McLellan Creek forks. However, Little Squaw Creek, and possibly the lower reaches of Big Squaw Creek, McLellan, and Nugget Creek where they extend into the Lake Creek Valley, were not scoured. Instead, these creeks were partially backfilled with glacial till when glaciers moved into the Lake Creek Valley. On Little Squaw Creek, glacial till containing exotic clasts overlies pre-glacial gravel as far upstream as the 2,800-foot elevation. Previous exploration, confirmed by the results of 2007 drilling, showed that the gold-bearing pre-glacial fluvial gravel in the lower reaches of Little Squaw Creek was not removed by the glaciers; rather, it is preserved under the till. Consequently, below the 2,800-foot elevation there are two distinct ages of placer deposits, the pre-glacial fluvial system and the overlying Pleistocene till and complex interglacial sediment deposition.

The glacial section in Little Squaw, Nugget, and Big Squaw Creeks includes, along with schist varieties seen in the fluvial gravel, a variety of clasts not found in bedrock within the district. Slate, marble, biotite- and muscovite-bearing granite, conglomerate, and quartzite are variably abundant, commonly as cobbles and boulders. A distinctive gray quartzite with abundant well-rounded black chert clasts is particularly abundant in the glacial till and is a good marker, along with the gray color of clay, for the glacial section. Clay is abundant in the glacial section. In some drill holes, clay layers up to 15 feet thick were encountered. Ice lenses up to 20 feet thick are present in the upper part of the glacial section on the left limit of Little Squaw Creek below about 2,350 feet elevation.

At the end of each Pleistocene glacial advance, gold-bearing placer deposits formed as channels cut into the top of the resultant lag of glacial till. The rich, gold-bearing gravel mined from the Mello Bench and gravel mined in the modern stream between the 2,500-foot and 2,700-foot elevations were in perched fluvial channels in the upper glacial section.

7.2.5 Mineralization

7.2.5.1 Lode Mineralization

Most high-grade mineralized zones within vein systems are less than 150 feet long and 2 to 10 feet wide. Vertical extents are unknown but probably exceed 200 feet. Individual prospects discovered at various points along major shear zones were originally thought to be discrete discoveries, but groups of prospects have since been determined to be related along a common structure. The Summit vein system, for instance, includes several other discoveries now believed related to the same system traceable for about 5,800 feet, over which mineralization is relatively continuous but varies widely from low-grade (0.20 parts per million [ppm] Au) to high-grade (>35 ppm Au). Over this strike length, the Summit system also spans a vertical range of 1,000 feet.

Some degree of lithologic control of mineralization has been suggested. Duke (1975) and Chipp (1970) believed gold mineralization may be preferentially hosted in carbonaceous phyllite and gray to black schist. There is an inferred lithologic control of the Chiga prospect that extends down dip and is tentatively projected to underlie the Summit footwall area, where it was intersected in RC drill hole SUM 12. At Aurora Gulch, differing lithologies host differing mineralization styles.

Most vein systems are closely situated within or in the adjacent hanging wall of the major shear faults, but there are exceptions. The Chandalar vein systems sort into two groups. The first group (e.g., Mikado, Eneveloe, Pioneer) are discontinuously mineralized major quartz veins also associated with subparallel gold-bearing lenses, parallel veins, and stringer and sheeted zones, within enveloping zones of alteration, shearing, and gouge. Because of the close proximity of deep-seated shear zones, the recurrent movement along these shears has consequently brecciated and deformed these vein structures. Alteration in gouge zones includes various clay minerals, predominantly kaolinite, black-to-green chlorite, granulated quartz, lesser albite, alunite, and carbonate as siderite and ferroan dolomite. Graphite is commonly associated with higher grade mineralization. Commonly, there is both banding and cross-cutting evidence of multiple stages of quartz precipitation, and each stage features varying mineralization, including gold as flakes and wires, arsenopyrite, pyrite, and accessory galena. Quartz veins with similar mineral assemblages, but without significant gold, are also found in the district and occur in the same vein system as mineralized quartz.

The second group of vein systems (e.g., the Little Squaw, Crystal, Grubstake East and West, the Jackpot, and perhaps the Star) is found in east-west-trending fractures in relatively close proximity to the major shear zones, but more distal than those described above. This type of deposit occurs in subparallel splay faults or fractures that horsetail off the major shear zones. Similarly, the distal segments of the Summit and Pioneer shear-hosted mineralization become more vein-like as the systems verge slightly (2 to 5°) from the main shear structure. There is little evidence of major post vein movement on the host fractures and no significant gouge development; however, smaller-scale recurrent movements result in banded- to ribbon-pattern laminae and slickenside within the veins. These veins have sharply defined footwall and hanging wall contacts with minor wall rock alteration, and quartz readily breaks free of wall rock. They have more continuity than the shear-hosted type but generally lack the enveloping alteration and associated low-grade auriferous zones. High-grade gold mineralization is often concentrated in the fine-grained ribbon banded zones that may occur along either contact, but the high-grade gold tends to favor softer wall rock of carbonaceous phyllite. Such zones can occur as ore shoots. Gold in ribbon banded quartz occurs as wires and flakes commonly up to several millimeters in size. These veins generally will also include wider bands of massive lower-grade, coarser grain quartz. Quartz veins with similar mineral assemblages, but without significant gold, are also found in the district and occur in the same vein system as mineralized quartz. For instance, the Grubstake West veins are excellent examples of composite veins with scorodite-stained ribbon banding on the footwall, yet samples (to-date) contain no significant gold values.

Some prospects, such as the Bonanza-Jupiter and the Kiska, appear gradational between the two vein styles. Similarly, the distal segments of the Summit and Pioneer shear-hosted mineralization become more vein-like as the systems verge slightly (2-5°) from the main shear structure. For example, the Pioneer exhibits highly sheared lenses of vein quartz where the vein system crosses over the shear but becomes distinctly vein-like at the Grubstake East prospect about one mile to the west and 200 feet north of the shear zone.

When first explored in 1909, the Little Squaw vein was estimated to have a small resource of 2,000 tons grading 1.55 oz Au/ton within a high-grade shoot of auriferous vein quartz containing visible gold (resource given as a non-compliant historic reference only). The shoot is exposed at the surface and at the 100 Level. Uncertain records account for no more than 625 oz of Au production. Ore would have been included with Mikado ore going to the Tobin mill. Workings include two levels, each about 300 feet long, connected by a winze and a 76-foot raise to the discovery outcrop. In 2006, ten reverse-circulation holes were drilled to explore the known and suspected side veins.

Quartz vein mineralization is localized in an ore shoot along a south-dipping fault on the 100 Level, where gold in the shoot is confined to the footwall zone of a composite vein. A 9- to 12-inch banded ribbon gold-quartz zone commonly contains 50 or more ppm Au as well as disseminated and thin seams of arsenopyrite, mica, scorodite, pyrite, and trace galena (Photo 7-1). Slickenside is common on many of the laminar planes that form the banding. Small clots of wire gold occur in vugs and on band surfaces and are very loosely attached to the rock. Veins can be traced westerly about 1,800 feet from the 100 Level and are open beyond that. On the east slope to Gold Creek, float quartz with visible gold was found. The Little Squaw vein zone is projected along a 110° strike eastward to the Crystal prospect, about 1.0 mile east. The vein at the Crystal prospect closely resembles the ribbon-banded 100 Level vein.

Two or possibly three veins are present at the Chandalar mine; the principal veins are the 100 Level vein to the north and a south vein about 125 feet south of the 100 Level vein. Ten holes were drilled in 2006 from six sites along the Little Squaw structure to test the known vein system at depth. Reconstruction of fault movement suggests that auriferous intercepts in 2006 (holes LS-2, 4) and intercepts in the 1982 drill holes correlate to the 100 Level vein; however, several fault orientations complicate interpretation. The best intercept was Hole LS-2, which cut 4.21 ppm Au over 20 feet, including 5 feet of 10.75 ppm Au. The1982 hole LS45N reported 0.46 oz Au/ton on another 100 Level vein 10 foot-intercept. The south vein was found in the 1982 holes 45S and LS3 and was also cut by 2006 holes LS-5 and -36.

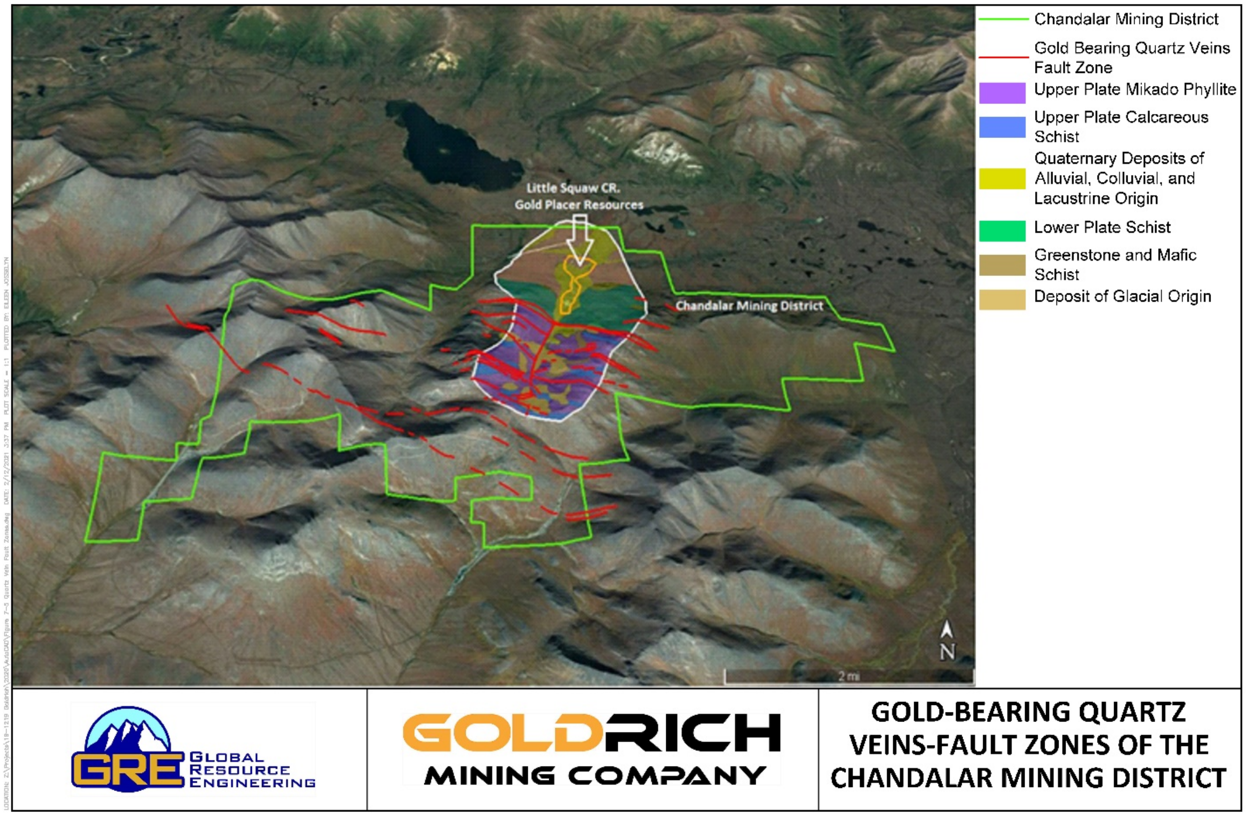
It is apparent from the combined 1982 and 2006 drill data that the vein on the 100 Level and the south vein may be mineralized over a longer strike length and depth than previously known, but the structure is highly complicated by offset faults.

Two drill holes at the Crystal in 2006 failed to intercept the vein seen in outcrop; however, hole CRY-30 did cut 35 feet of low-grade, steeply dipping sheeted veining. Flat-lying faults are believed to have displaced the Crystal vein.

Photo 7-1: 100-Level Ore Shoot of Little Squaw Quartz Vein, Looking West

This series of gold-bearing quartz veins-fault zones (red) are likely the source of the gold within the placer deposits in Little and Big Squaw Creek catchment (Figure 7-5). GRE believes these quartz veins have excellent potential to host a significant lode deposit. The larger placer deposit in the Little Squaw basin indicate more gold mineralization is likely to be found in the Little Squaw valley.

Figure 7-5: Gold-Bearing Quartz Veins-Fault Zones of the Chandalar Mining District with Geology Map of the Chandalar Catchment



7.2.5.2 Placer Mineralization

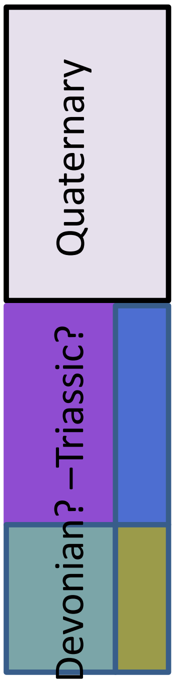
Placer gold occurs within the pre-glacial fluvial, interglacial glaciofluvial, and post-glacial fluvial, permafrost deposits on Little Squaw Creek. The gold is coarse, crystalline, and bright, indicating that it was transported only a short distance from its sources. Small nuggets are common, and there is little fine-grained gold (-80 mesh). Quartz inclusions and attachments are common on gold particles but make up only a few percent by volume.

The pay streak on Little Squaw Creek is subdivided into the “canyon” placer upstream of a bedrock constriction underlying Lines 4.8 through 6, and an “alluvial fan” placer downstream of the buried constriction that extends at least as far north as Line 1.2., a distance of about 2,000 feet. Fluvial gravel hosting the placer is mostly composed of gray to black chloritic schist, which is commonly seen in the surrounding hills.

Gold-bearing gravels composing the alluvial fan range from 15 to 137 feet thick and average 80 feet thick over a width of up to 1,262 feet. The pay streak within the fluvial gravel on the canyon section varies from 50 to 136 feet thick over a width from 240 to 570 feet. The placer deposit is open to further mineralization to the east, west, and north. Figure 7-6 shows the stratigraphic column.

Figure 7-6: Goldrich Stratigraphic Column

Stratigraphic Column of the Chandalar Mining District



Unconsolidated Deposits

Alluvium, alluvial fan, placer mine tailing, alluvial terrace, colluvial, lacustrine deposits, and glacial till and outwash

Bedrock Deposits

Upper Plate Units

Undifferentiated bedrock, meta-igneous greenstone, phyllite, schist, meta-turbitite schist, meta-sandstone and quartz-mica shist, and muscovite-schist

Lower Plate Units

Meta-volcanic agglomerate, meta-igneous greenstone, tuffaceous schist, meta-mudstone schist, mica quartz schist, and metamorphosed felsic tuff

Other heavy minerals occur in sparse to minor amounts and include, in general order of abundance: magnetite, magnetic pyrrhotite, locally abundant pyrite, hematite/goethite, ilmenite, scheelite, and galena with trace amounts of garnet, stibnite, and arsenopyrite.

Fine galena cubes are relatively abundant in drill samples recovered from bedrock and overlying gravel on Line 1.2, Hole 1. This may present a lode exploration target.

Overburden on the fan is composed of frozen, clay-rich glacial till. Ice is moderately abundant in the overburden of the alluvial fan and may compose 10% of the total volume. Ice is particularly prevalent under the western end of Line 1.2, where it forms massive lenses 15 to 20 feet thick in Hole 18. Thawing ice appears to be responsible for the actively subsiding thermokarst pond east of the road. The barren till overburden averages 65 feet thick. Overburden on the canyon placer is mixed glaciofluvial sediments and can carry minor concentrations of gold at the surface and disseminated throughout. The overlying barren sections have a highly variable thickness, ranging from 0 to 75 feet with a 41-foot average.

8.0 Deposit Types

Portions of this report section are adapted from Barker, et al. (2009), Mendham et al. (2018), Gillerman (2012), Kuehner (2013), Rasmussen (2013), and Walter (2020). The author has reviewed this information in detail and finds it reasonably accurate and suitable for use in this report.

At Chandalar, both lode (Section 8.1) and placer (Section 8.2) gold exist within the district. The placer deposit is the focus of this report, but a significant exploration potential exists to find and develop the lode deposits that were the source of the placer deposit.

8.1 Lode Deposits

Lode gold occurs throughout the Chandalar property in definable systems of veins, veinlets, disseminations, and auriferous lenses of quartz within or adjacent to northwest-trending shear zones. Figure 6-1 shows the location of all known lode prospects and placers with past production.

For more than a century, prospectors have explored the Chandalar property for high-grade gold-quartz veins. Numerous veins have been found, and several have seen minor production. The largest historical mine was the Mikado mine which produced 6,700 oz Au from 10,500 tons at a head grade of 0.99 oz Au/ton. The schist-hosted Mikado mineralization occurs in discontinuous elongate lenses and veins that pinch and swell as border zones of stockwork and sheeted veinlets. The Mikado shear zone, generally found in the hanging wall, contains the majority of the mineralization. Recurrent movement along the shear explains the discontinuous nature of the mineralization, the pulverization of quartz to a clayey sugary texture, and the numerous ragged inclusions of host rock.

Elsewhere, veins such as the Little Squaw, Crystal, Kiska, Prospector, and Grubstake are hosted in splay faults and are more competent and continuous, but gold values still tend to be discontinuous, and mineralization primarily occurs as pods and ore shoots. Overall, auriferous and subparallel quartz veins of the Chandalar district occur across a five-mile width and a northwest-trending four-mile strike length (see Figure 8-1).

8.1.1 Differing Theories of Ore Genesis

Studies are not conclusive as to the ore genesis present in the Chandalar lode deposits. Despite early theories that the deposit could be epithermal, subsequent geologic investigation has the lode deposits narrowed down to either an orogenic or magmatic gold mineralization system or a combination of the two.

Dr. Samari's review of all available data and documents indicates that it is impossible to consider only one of the orogenic or magmatic models for the gold mineralization with high certainty at Chandalar district due to existing evidence for two models.

8.1.1.1 Evidence of an Orogenic Model

Dillon and others (1989) described the placer and lode deposits in the upper Koyukuk and Chandalar Districts. In their studies, they noted the presence of abundant stibnite at Koyukuk but more abundant arsenopyrite at Chandalar, along with anomalous tungsten (W), molybdenum (Mo), and mercury (Hg). They speculated that the gold was remobilized (in Cretaceous times) from Devonian granite cupolas in the region. This geochemical signature of Sb, As, W, and Hg in late gold quartz veins is somewhat reminiscent of mineralization at the Stibnite District in central Idaho. Gold ore at Stibnite is located in a late orogenic shear and fault zone near horsetail and cymoid loop structures. In summary, Dillion found evidence that the age, mineralogy, host rocks, and geochemistry are compatible with an orogenic gold model.

Goldfarb and others (2005) provided a comprehensive review of these “epigenetic gold deposits in metamorphic terranes,” which they noted typically form during the late stages of orogeny and may also be called “orogenic gold deposits.” Archean greenstone hosted deposits are one type of orogenic gold deposits but will not be considered further in this discussion. More relevant to the Chandalar District are a few gold-quartz vein orogenic deposits that are hosted in metamorphosed sedimentary rocks within Phanerozoic fold belts. These seem more similar and analogous to the samples, structure, and geologic setting of the Chandalar District.

Petrographic studies were carried out by Gillerman (2012) on eight HQ drill core samples from the property support an orogenic gold model. Gillerman believed mineralogy, host rocks, timing, and geochemistry are compatible with an orogenic deposit, though she noted in her report “the exact origin of the gold is unknown without more study.”

8.1.1.2 Evidence for a Magmatic Model

Lamal (1984) carried out the fluid inclusion that provides data regarding the homogenization temperature during mineralization. The fluid inclusion study showed evidence of a magmatic genesis for the Chandalar district. A summary of Lamal’s work is shown below:

Five samples from quartz veins at the Chandalar district were analyzed and fluid salinity was calculated from the depression of the H_2O melting temperature in this study was from 3-6 % NaCl. Furthermore, this study shows that the homogenization temperature for four samples is from a minimum of 252° to a maximum of 320° degrees. This temperature range is consistent with a magmatic ore genesis model. Only one sample shows the lower range of homogenization temperature, which is 207-285°. As a summary of fluid inclusion study, it was found that gold-bearing quartz veins in the little Squaw are crystallized at about 275° and 825 bars with fluids containing an average of 0.18 mole % CO2. Although Lamal (1984) mentioned “such an extreme gradient would only be encountered in areas of igneous activity, and, since there is no evidence of igneous activity at the time of mineralization in this part of Alaska, then it is concluded that boiling and consequent mineral deposition took place when lithostatic conditions switched to hydrostatic conditions after the peak of metamorphism during uplift", Based on newer studies that have been carried out so far, which present hereafter, the fluid inclusion study by Lamal (1984) emphasizes that the quartz veins are mesothermal deposits.

Rasmussen (2013) added to the magmatic genesis theory of the lode gold quartz veins at Chandalar based on the following:

- high-power electron microprobe microscopy at the University of Washington
- oxygen isotope method at Washington State University and
- data from radiometric/magnetic airborne surveys over Chandalar

Microprobe studies is another method of identifying genetic sources of lode deposits. Microprobe study by Kuehner (2013) on six core samples was reported by Rasmussen to show several characteristics consistent with magmatic ore genesis. All sampled sections contain accessory phases rich in incompatible elements, such as Zr, P, Rare Earth Elements (REE), Th, U, Y, F and Ti. These elements are hosted in phases such as monazite, zircon, baddeleyite, F-apatite, thorite, W-rich rutile, and possibly xenotime. Backscattered electron imaging used in conjunction with wavelength and energy dispersive spectroscopy was employed. Using these tools, it is clear that the euhedral W-zoned rutile grains in DDH20-747 coprecipitated with the sulfides. Inclusions of a variety of these incompatible element rich phases, including thorite, in the sulfides of sample DDH41-256 confirms the coprecipitation of these phases with the sulfide linked to granitic magma source.

Rasmussen (2013) also described that the oxygen isotopic study results from Washington State University reflecting a magmatic source for lode gold-quartz veins in the Chandalar district. GRE's evaluation of the oxygen isotopic study Larson (2014) prepared at Washington State University, shows Chandalar quartz is more associated with metamorphic rocks and metamorphic water instead of granitic rocks. A magmatic model must have a pluton, but to date there is no drilling evidence of a pluton in the region. Rasmussen mentions an aeromagnetic survey and map from the area prepared by USGS MF 878C (1956) that clearly shows a magnetic anomaly just in the south and beneath of the Chandalar Mine. GRE agrees that this anomaly has a characteristic ring structure indicative of an intrusive body (pluton).

In 2012, Richard Walters recognized that the Chandalar load deposits have intense quartz-carbonate hydrothermal alteration of pegmatitic dikes. He described these dikeletes as follows:

- All dikelets encountered in the drilling are mineralized. None of the wall rocks, unless completely chloritized, are.
- Most of the pegmatites are not associated with fault structures, but a few are. These are unlike the mineralized quartz veins which seemingly all occur as multiple sheared (seven or more episodes) and boudinized pockets along major NW faults where intersected by NE structures over the sub-regional NE oriented airborne magnetic anomaly.
- Three cross cutting phases of quartz veins and two phases of cross cutting dikelets were recorded. The injection timing sequence of the quartz veins verses the dikelets - prior, coeval or later - was not known.

Associated with these mineralization events along the dikelets, Walters mentioned Au-bearing quartz-feldspar-muscovite-kaolinite-siderite veins with monazite, xenotime, thorite all of which suggest a magmatic contribution to the mineralizing hydrothermal fluids at Chandalar.

In October 2020, Richard Walter noted:

> "because of our exploration considering structural geology, petrographic, geochemical and geophysical evidence we have realized that all of the gold is sourced within a system of magmatic hydrothermal alteration features (like small pegmatitic dikes and chloritized schist). We believe these features are common to and link all the hard-rock (lode) prospects (the weathering of which generated the gold placer deposits), and furthermore are an outlying expression of an underlying

gold bearing pluton. Goldrich is currently defining drilling targets for a hard-rock gold deposit in an area of interest approximately 1,800 feet wide and over five miles long, possibly underlain by a series of mineralized magmatic intrusions (plutons)."

8.1.2 GRE's Opinion Based on the Existing Data

Dr. Samari believes existing data for the area supports both models of the orogenic and magmatic gold mineralization for non-placer deposits.

Therefore, Dr. Samari surmises that there could be at least three different timing sequences of gold mineralization including:

1. At least one phase of mineralization before the Devonian (before the pluton's intrusion)
2. One phase associated to the intrusion
3. A post-intrusion mineralization phase

In 2008, Richard Walters mentioned that disseminated gold mineralization in schist boulders found on St. Mary's Pass prospect. This is strong evidence for a nearby mineralization even before the Devonian period (see No.1 above).

In this scenario, gold mineralization in Proterozoic to lower Paleozoic metasedimentary schist and greenstones might have an orogenic source (see No. 1 above) while the gold-bearing quartz veins and gold-bearing pegmatite dikeletes have magmatic source (see No. 2 and 3 above).

For example, it appears that pegmatitic dikelete (which crosscut schists) are associated with the first phase of intrusive activity (No. 2 above). During a later phase of intrusive activity (No. 3 above), hydrothermal fluids caused silica and chlorite alterations. These two alterations synchronously were the source of both Au-bearing cross-cutting quartz veins and Au-bearing pegmatite dikeletes.

In summary, the preponderance of the data from exploration to date support interpretation of the Chandalar veins and dikelets as metasediment-hosted, magmatic, low-sulfide mesothermal deposit. If this is the case, this theory places Chandalar in a category with other important gold deposits of the Yukon-Alaska territory such as Fort Knox (8 M oz.) and Donlin Creek (25 M oz.). Therefore, it is possible that Chandalar's vein lode deposits may be an expression of a deeper, larger gold system worthy of substantial exploration.

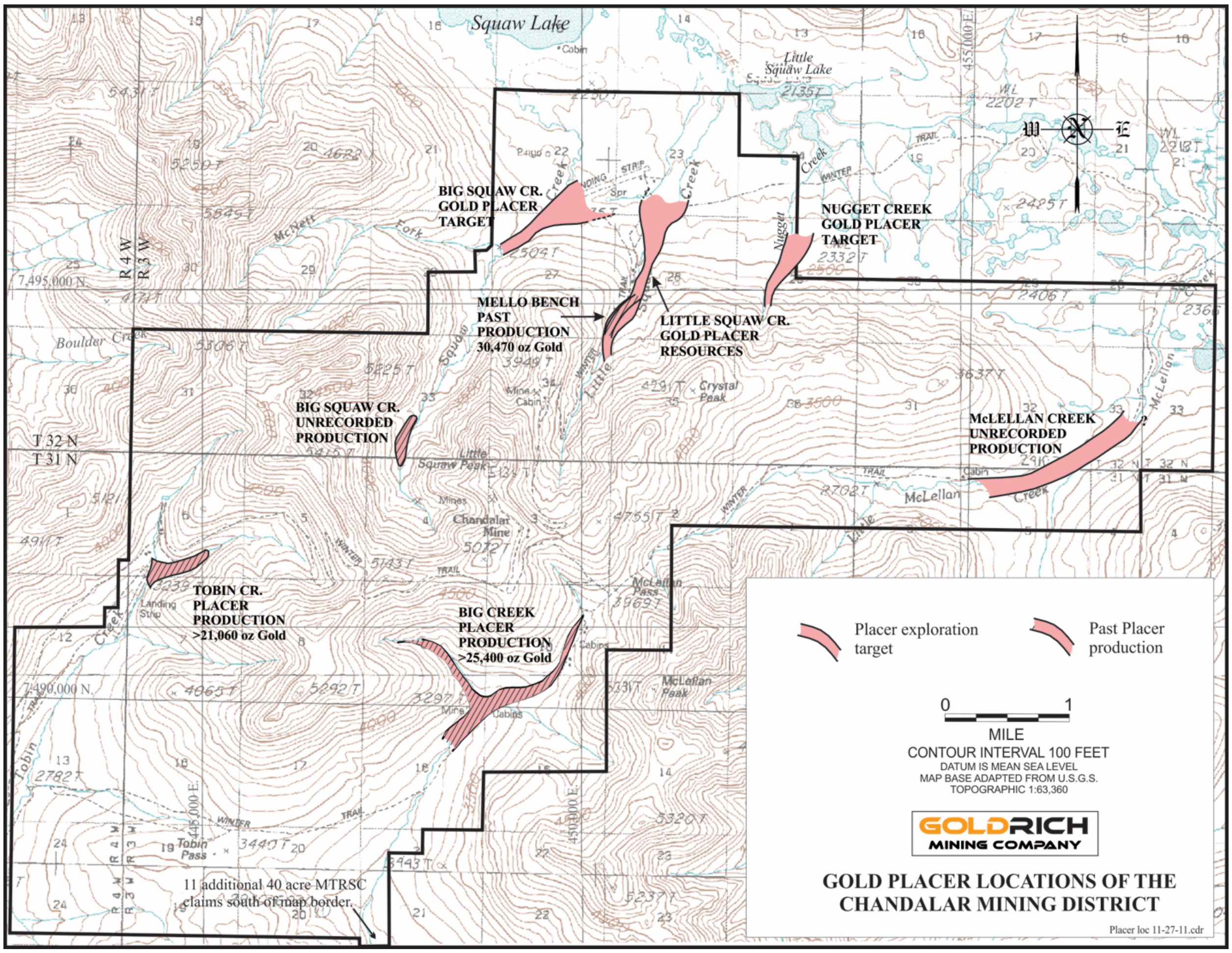
8.2 Placer Deposits

Placer gold in the Chandalar Mining district was liberated from lode sources of the former highland weathering surface that rose more than 3,000 feet (914 meters) above the surrounding lowlands during several episodes of erosion and concentration and was further complicated by repeated advances of Quaternary glaciation from the north. Subsequent downcutting on each glacial retreat occurred in response to newly established base levels along the wide valleys to the north and west of the district. These glacial events scoured the upper placer streams of Tobin, Boulder, Woodchuck, McLellan, and Big Squaw Creeks but did not destroy pre-glacial pay streaks at the lower valleys of Little Squaw, Tobin (above

Woodchuck Creek), and probably not Big Squaw and McLellan Creeks. These pre-glacial placer deposits were preserved during glaciation, and significant targets remain (Figure 8-1).

The complex glacial-fluvial history of the Chandalar region formed placers and placer exploration targets that exhibit several differing deposit types. On the north and east side of the district, the placers are a combination of stacked sequences of normal fluvial and glaciofluvial channel deposits that have formed in second order streams and feature bedrock and false bedrock pay channels. On Little Squaw Creek, and possibly on Big Squaw and Nugget Creeks, resistant bedrock along the north margin of the hills has influenced rapid stream gradient changes that created ancestral canyons below which paleo-fluvial and modern glaciofluvial fans have formed.

Figure 8-1: Gold Placer Locations of the Chandalar Mining District



The glacial aspects of several of the Chandalar placer deposits, notably Chandalar Mine, where drill data are now available, can be approximately compared to the glaciofluvial deposits in the Valdez Creek (south-central Alaska) (Reger, et al., 1990), Porcupine (southeast Alaska), and the Kolyma (Russian Far East) placer gold districts. The Valdez Creek deposit yielded 514,000 oz Au from glaciofluvial and fluvial gravels, including both bedrock and perched auriferous channels as deep as nearly 200 feet.

The Little Squaw placer model is viewed as:

- aggradational pre-glacial channel, terrace, and fluvial fan deposits
- intra- glacial and post-glacial aggradational, stacked, truncated channel-segments that composite into placer deposits
- most of the deposit is permanently frozen (permafrost).

The later type is formed by combined fluvial and glaciofluvial processes and is deposited on glacial false-bedrock. These processes result in deposition of significant thickness of auriferous sediment as exhibited at Little Squaw Creek and possibly within other drainages along the north side of the district.

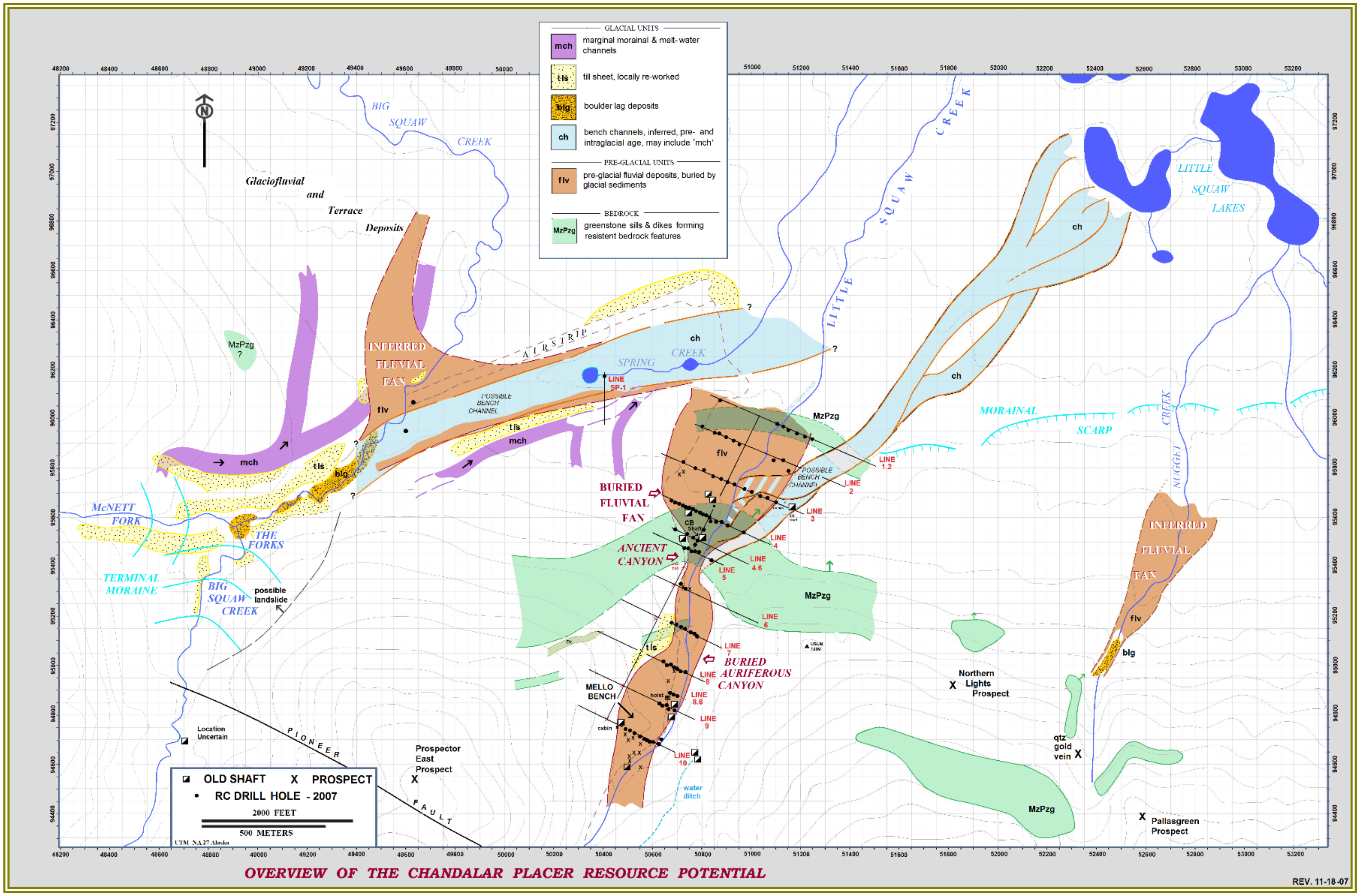
The distribution of Coldfoot sub-terrane rocks as favorable auriferous source rock and various historical accounts of placer gold suggest this general model should also be the focus of exploration at McLellan, Nugget, and several small creeks southeast of the Company’s property that are tributaries to Middle Fork, including Rock, Tribley, Day, Dictator, and Agitator Creeks.

Little Squaw Creek is a second-order stream with gold concentrated in placers in its upper reach and in a wider alluvial fan placer where it exits the canyon (Figure 8-2).

This deposit is geologically characterized as an aggradational placer gold deposit. It is unusual in the sense that it is the only such known alluvial, or placer, gold deposit in Alaska, although many exist in Siberia. Goldrich’s discovery contrasts to others in Alaska that are commonly known as bedrock placer gold deposits. Aggradational alluvial gold deposits contain gold particles disseminated through thick sections of unconsolidated stream gravels in contrast to bedrock placer deposits where thin but rich gold-bearing gravel pay streaks rest directly on bedrock surfaces. Aggradational placer gold deposits are generally more uniform and thus more conducive to bulk mining techniques incorporating economies of scale. This contrasts with bedrock placer gold deposits where gold distribution tends to be erratic and highly variable. The plan view of the Chandalar Mine deposit is somewhat funnel-shaped, and because of this has been divided into two distinct geomorphological zones: the Gulch, or narrower channel portion, and the Fan, or broad alluvial apron portion.

Similar geologic settings are inferred at other northeast-flowing streams draining the district. A paper by Russian placer geologist Yuri Goldfarb (2007) proposes to reclassify placer deposits found in northern latitudes based on source, morphology, and dynamic (lithomorphodynamic) processes that influence the character of placer gold particles and placer gold deposits. His classification can be extrapolated to generalize about size and grade and, thus, to economic parameters of placer deposit classes. A large body of field data from exploration and mining projects, mostly Russian, but some Alaskan, served as his basis. Under his scheme, the Little Squaw placer deposit would be classified as an aggradational placer due to its close proximity to source rock, steep gradient, high variability of gold grain size, great thickness of gold-bearing gravel, and stacked, complex lithofacies with an overprint of an erosional placer, which is actively eroding through the aggradational placer. Examples of this type of placer deposit where productive mining has occurred include the Hogum, Osceola, and Manhattan placers of Nevada (Vanderberg, 1936) and the Greater Kuranakh placer in the Aldan gold province of north-eastern Russia (Goldfarb, 2007).

Figure 8-2: Overview of the Little Squaw, Big Squaw, and Nugget Pre-glacial Fluvial Deposits Buried by Glacial Sediments



GLOBAL RESOURCE ENGINEERING

9.0 Exploration

Portions of this report section are adapted from Barker, et al. (2009) and Mendham et al. (2018). The authors have reviewed this information in detail, finding it reasonably accurate and suitable for use in this report.

Since 2003, in addition to drilling programs, as detailed in Section 10.0 *Drilling*, exploration activities carried out by Goldrich include:

- local mapping of about 40 identified prospect areas
- collection and geochemical analyses of approximately 1,350 soil, 1,305 rock, 67 stream sediment, and 11 water samples, and preparation of anomaly maps
- a trenching program of 45 trenches consisting of 5,937 feet, of which 4,954 feet was exposed bedrock, and collection of about 550 trench-wall channel samples
- ground magnetometer survey grids of 15 prospect areas and survey lines totaling 28 miles.

To date, Goldrich has collected and assayed a total of 3,431 surface samples and analyzed approximately 4,500 drill samples.

In addition, two airborne geophysical surveys have been completed. In 2011, approximately 770 line-miles (1,246 line-kilometers) were flown by an international geophysical contractor over the entire Chandalar property along flight lines 100 meters apart. Preliminary magnetic data revealed known mineralized structures with good clarity and, more importantly, identifies sharp new prospect-scale and district-scale anomalies and mineralized trends.

In 2014, Goldrich completed another airborne radiometric and magnetic survey, also approximately 770 line-miles (1,234 line-kilometers) to test an intrusion-related model for emplacement of lode quartz-gold occurrences. Results of the airborne study demonstrate a broad northwest-trending belt of elevated potassium values with a centrally located, kilometer-scale feature where thorium values are elevated relative to potassium. The potassium/thorium feature anomaly is closely associated with magnetic anomalies to form a circular kilometer-scale feature in the highlands above and adjacent to the Little Squaw Placer deposit consistent with an intrusive body at depth and is central to the northeast-trend of lode quartz-gold occurrences.

9.1 Pan Sampling

Reconnaissance exploration on Chandalar Mine was carried out by collecting pan samples from dumps associated with old prospect and production shafts, and from sites within old open cuts. Sample locations and site characteristics were recorded at each sample site, and weighable amounts of gold recovered from the pans were saved and weighed. The results of the pan sampling indicate widespread occurrence of potentially economic amounts of placer gold. Surficial evidence (relief and projection) suggests the possibility of buried fluvial channels pre-dating most recent glaciation or ice-marginal channels during glaciation, draining to the northeast from Little Squaw Creek.

9.2 Trenching

In 2006, a limited trenching program was performed to verify the results of two holes drilled at Chandalar Mine by Daglow Exploration, Inc., in 1997. Each of the drill holes (LS97-7 and -11) encountered gold within 20 feet of ground surface. A trench was excavated to 17 feet adjacent to each drill hole collar to sample the reported gold-bearing intervals. Trench samples measuring one bank cubic foot were panned in a nearby stream, and the results confirmed the presence of placer gold at shallow depths.

Verifying the earlier drilling program results by trenching suggests that drilling the deeper placer deposits can yield reliable exploration data. The 2007 drilling program did locate valuable placer gold deposits at Chandalar Mine in perched pay streaks concentrated on clay-rich “false bedrock” within the fluvial gravel section and variably enriched deposits of gold in deeper pay streaks found on bedrock. Minor, probably sub-economic, amounts of placer gold were found in erratic locations within the overlying glacial sediments.

10.0 Drilling

10.1 Hard-Rock (Lode)

10.1.1 Reverse Circulation Drill Program

In 2006, Goldrich completed a reverse circulation drill program consisting of 39 drill holes for 7,763 feet (2,366 meters) on nine of some thirty gold load prospects (Little Squaw Mining Co., 2006). The results from four prospects, Little Squaw, Summit, Eneveloe and Ratchet Ridge, showed the need for follow-up diamond core drilling. The results included a quartz vein intercept on the Eneveloe Prospect of 25 feet of 5.85 grams per tonne (g/t) gold (0.171 ounces per ton [oz/ton]), including 5 feet of 25.40 g/t (0.742 oz/ton) gold.

10.1.2 HQ Core Drill Program

In 2011, Goldrich completed a 25-hole, 14,444-foot (4,404-meter) exploratory program, using HQ size core, tested six prospect areas (see accompanying map) located along a 4-km (2.5-mile) long northeast trending belt of gold showings. The drill cores contain a total of 56 mineralized intervals of 0.5 or greater grams per tonne gold (g/t Au) that average 2.3 meters (7.5 feet) in length and have a weighted average grade of 1.66 g/t Au (see accompanying table). Gold-bearing intercepts were obtained in 72% of the holes, with many having multiple intercepts.

10.2 Placer

10.2.1 Reverse Circulation Drill Program

In 2007, Goldrich discovered and partially drilled out a large placer gold deposit in the Little Squaw Creek drainage with a conventional, air-rotary reverse circulation (RC) drilling program targeting bench deposits at Chandalar Mine with 101 drill holes. Drilling in the 101 valid drill holes totaled 14,856.5 feet. The program included four exploration holes on Spring Creek and two on Big Squaw Creek, and all together totaled 15,303.5 feet of drilling. Drill lines were on 500-foot spacing, with holes 50- to 100-feet apart. All holes were drilled vertically into horizontal placer targets, and all intercepts are considered representative of the true thickness of mineralization. A plan map showing the location of drill lines and drill holes at Chandalar Mine is presented as Figure 10-1, and the results of the drilling are summarized in Table 10-1.

Of the 107 holes collared, 87 were completed to bedrock, and 20 holes were either abandoned due to ground caving or swelling or were terminated at the full extent (210 feet) of the available drill rod without reaching bedrock. All holes were drilled "open," without casing.

In 2009, as a result of the 2007 placer drill program, Goldrich opened the Chandalar Mine as a test project. Favorable results from the test project led to the expansion of the mine in 2010. Total production from 2009 to 2018 was approximately 44,209 ounces of fine gold.

10.2.2 Sonic Drill Program

In 2013, 72 sonic dill holes were completed, of which 11 have no assay data, to infill the 2007 drilling program to a line spacing of 250 feet, with one line infilled to 125 feet. Drilling in the 61 valid drill holes totaled 6,253.7 feet (Figure 10-2).

In 2017, 230 sonic drill holes totaling 14,303 feet were drilled. The drill lines were spaced at 125 to 250 feet on and around the southwestern canyon portion of the deposit (Figure 10-2).

Figure 10-1: Placer Geology and Drill Hole Locations in 2007 at Chandalar Mine

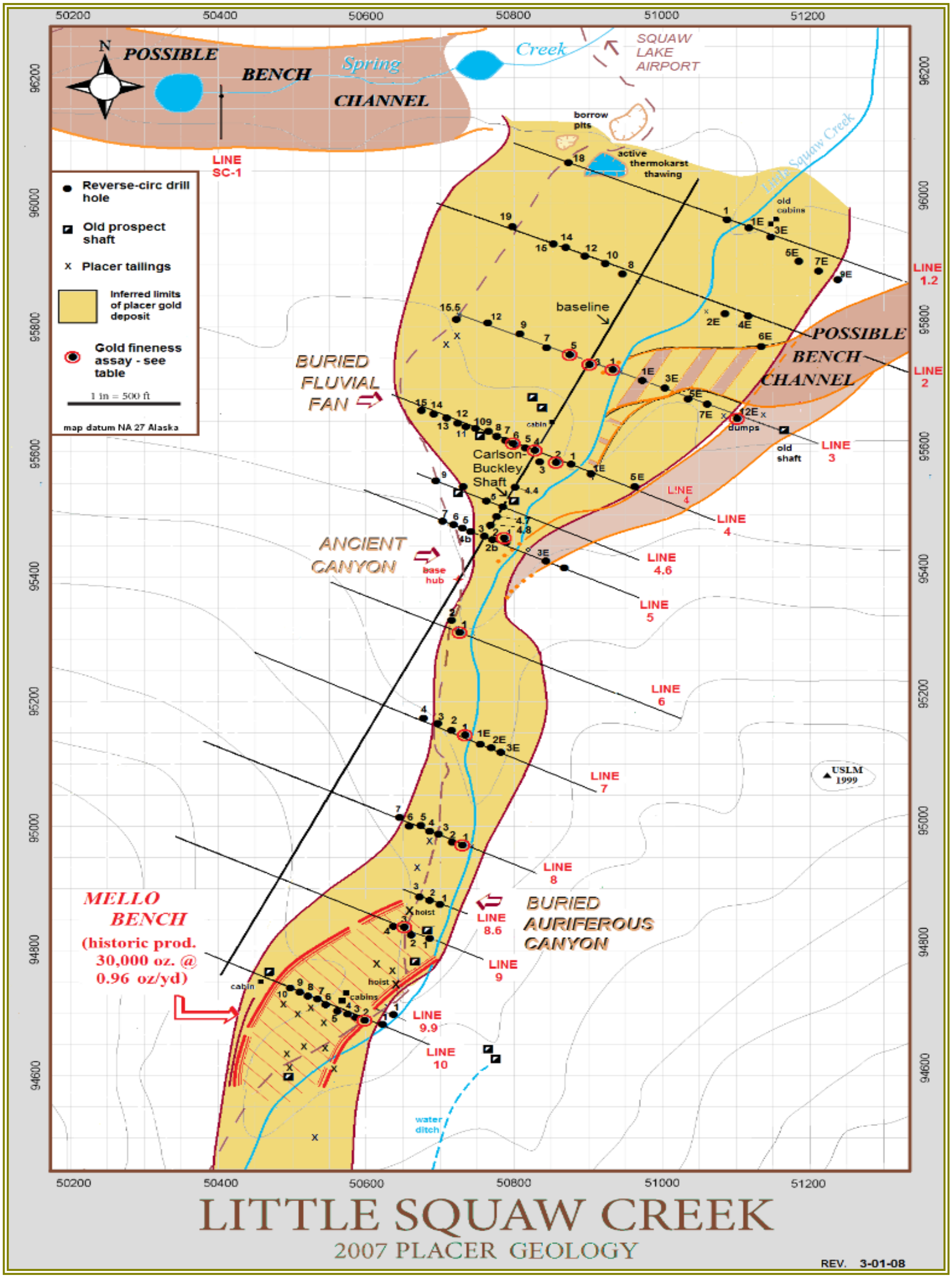


Table 10-1: Results of 2007, 2013, and 2017 Placer Drilling

Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
2007 Drill Holes					
L1.2H1	65964.64	48303.92	2310.62	135	RC
L1.2H18	65200.38	48686.35	2323.7	95	RC
L1.2H1E	66058.51	48256.83	2309.96	145	RC
L1.2H3E	66152.67	48215.68	2308.23	135	RC
L1.2H5E	66247.66	48181.12	2304.86	115	RC
L1.2H7E	66336.02	48138.15	2304.18	105	RC
L1.2H9E	66429.55	48096.6	2304.39	105	RC
L2H8	65476.5	48071.48	2341.05	210	RC
L2H10	65388.75	48130.36	2344.29	210	RC
L2H12	65302.39	48154.78	2352.41	210	RC
L2H14	65206.48	48204.85	2353.73	206	RC
L2H15	65161.94	48228.61	2354.36	210	RC
L2H19	64975.83	48300.1	2362.57	210	RC
L2H2E	65930.67	47866.03	2335.08	130	RC
L2H4E	66035.45	47861.15	2331.36	110	RC
L2H6E	66087.82	47703.03	2350.48	90	RC
L3H1	65442.46	47566.54	2366.96	165	RC
L3H3	65342.61	47596.64	2383.93	185	RC
L3H5	65250.93	47627.69	2385.71	180	RC
L3H7	65163.65	47656.81	2384.31	180	RC
L3H9	65049.55	47727.55	2384.64	200	RC
L3H12	64922.94	47758.17	2390.78	210	RC
L3H15.5	64744.11	47799.14	2403.86	210	RC
L3H3E	65587.75	47513.67	2368.14	155	RC
L3H5E	65681.36	47476.34	2373.52	145	RC
L3H7E	65777.4	47427.88	2386.19	135	RC
L3H9E	65865.25	47401.67	2387.46	120	RC
L3H12E	66005.62	47346.32	2388.53	105	RC
L4H1	65276.45	47057.82	2413.55	153	RC
L4H2	65233.18	47078.17	2414.22	150	RC
L4H3B	65164.18	47070.95	2416.74	160	RC
L4H4	65135.73	47116.12	2419.18	159.5	RC
L4H5	65090.42	47133.52	2420.07	180	RC
L4H6	65040.55	47150.34	2426.43	180	RC
L4H7	64999.39	47171.88	2428.81	195	RC
L4H8	64944.86	47188.19	2429.63	210	RC
L4H9	64907.88	47206.89	2430.2	210	RC
L4H10	64858.05	47225	2429.77	210	RC
L4H11	64802.74	47226.06	2432.25	155	RC
L4H12	64765.35	47261.06	2433.86	175	RC
L4H13	64718.73	47280.98	2436.96	170	RC
L4H14	64667.76	47296.9	2438.78	180	RC
L4H15	64617.99	47315.52	2440.08	165	RC

Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
L4H1East	65371.2	47020.8	2408.6	95	RC
L4H5East	65559.34	46951.33	2444.45	90	RC
L4.4H3	65049.59	46940.24	2435.42	195	RC
L4.6H3	65007.11	46844.64	2451.3	190	RC
L4.6H5	64918.59	46879.99	2455.37	125	RC
L4.6H7	64829.41	46941.39	2451.13	95	RC
L4.6H9	64701.03	46967.16	2452.04	85	RC
L4.7H3	64978.14	46795.57	2449.33	210	RC
L4.8H3	64961.36	46751.04	2454.54	210	RC
L5H1	65014.98	46632.93	2451.77	43	RC
L5H1B	65014.99	46645.86	2451.37	135	RC
L5H2	64962.03	46648.74	2457.87	55	RC
L5H2B	64962.39	46658.24	2456.79	120	RC
L5H3	64919.39	46667.87	2469.37	67	RC
L5H3B	64922.42	46662.69	2469.3	115	RC
L5H4	64876.54	46687.71	2473.13	68	RC
L5H4B	64869.28	46702.64	2473.2	100	RC
L5H5	64830.22	46706.59	2477	100	RC
L5H6	64776.65	46724.76	2475.2	67	RC
L5H7	64731.76	46740.87	2479.69	65	RC
L5H3E	65207.33	46557.53	2479.41	160	RC
L5H5E	65292.83	46516.1	2489.56	120	RC
L6H1	64896.28	46152.23	2494.48	135	RC
L6H2	64829.38	46157.95	2527.13	150	RC
L7H1	64871.57	45621.38	2554.55	105	RC
L7H2	64823.75	45639.87	2560.54	105	RC
L7H3	64754.4	45682.98	2585.38	130	RC
L7H4	64725.74	45682.39	2586.74	130	RC
L7H1E	64958.99	45585.81	2559.63	135	RC
L7H2E	65000.53	45564.18	2564.07	112	RC
L7H3E	65038.19	45529.86	2566.73	120	RC
L8H1	64915.94	45038.86	2626.82	150	RC
L8H2	64866.75	45055.92	2656.74	160	RC
L8H3	64804.46	45095.08	2670.14	190	RC
L8H4B	64761.39	45102.96	2682.45	205	RC
L8H5	64717.32	45121.88	2692.73	210	RC
L8H6B	64663.73	45113.22	2695.6	210	RC
L8H7	64630.15	45155.26	2695.36	210	RC
L8.6H1	64831.1	44716.05	2658.44	120	RC
L8.6H1B	64833.36	44709.54	2661.43	130	RC
L8.6H2	64785.52	44734.07	2666.55	105	RC
L8.6H3	64738.94	44748.55	2684.41	105	RC
L9H1	64775.54	44531.61	2684.44	120	RC
L9H2	64729.05	44546.82	2695.49	130	RC
L9H3	64691.59	44562.33	2704.35	145	RC

Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
L9H4	64638.96	44581.52	2719.29	145	RC
L9.9H1	64751.99	44195.42	2704.42	135	RC
L10H1	64598.21	44058.39	2724.93	100	RC
L10H2	64533.68	44078.61	2756.47	145	RC
L10H3	64477.52	44091.89	2772.72	160	RC
L10H4	64446.45	44128.46	2788.76	190	RC
L10H5	64407.97	44118.75	2792.39	175	RC
L10H6	64350.66	44154.08	2804.92	167	RC
L10H7	64298.48	44171.28	2818.14	200	RC
L10H8	64266.95	44195.7	2821.47	199	RC
L10H9	64230.49	44232.56	2824.3	90	RC
L10H10	64156.78	44220.67	2838.68	135	RC
L11H2	64121.33	43707.74	2812.15	130	RC
Total				**14,856.5**	
2013 Drill Holes					
L2H0	65830.63	47901.92	2336.71	124	Sonic
L2H2B	65739.54	47942.7	2339.59	150	Sonic
L2H8E	66152.55	47630.55	2353	80	Sonic
L2.5H0	65691.12	47702.37	2354.03	144	Sonic
L2.5H2	65600.04	47742.85	2355.24	153	Sonic
L2.5H2E	65783	47661.88	2351.54	140	Sonic
L2.5H4	65508.39	47783.74	2356.81	156.5	Sonic
L2.5H4E	65874.18	47620.67	2361.68	110	Sonic
L2.5H6	65416.99	47824.34	2358.12	153	Sonic
L2.5H6E	65965.53	47580	2366.66	110	Sonic
L2.5H8	65325.78	47865.39	2360.77	160	Sonic
L2.5H8E	66056.76	47539.74	2364.81	90	Sonic
L2.5H10E	66130.85	47527.05	2361	90	Sonic
L2.75H4E	65763.4	47569.7	2380	110	Sonic
L2.75H6E	65865.1	47523.77	2376	110	Sonic
L2.75H8E	65953.69	47484.4	2376	100	Sonic
L2.75H10E	66042.27	47441.75	2375	100	Sonic
L2.75H12E	66140.69	47399.1	2370	75	Sonic
L3H3EB(twin)	65573.26	47524.33	2368.14	155.2	Sonic
L3H11E	65937.1	47360.1	2387	105	Sonic
L3H14E	66074.89	47304.33	2384	110	Sonic
L3H17	64698.82	47877.4	2402	110	Sonic
L3H19	64613.52	47926.61	2405	110	Sonic
L3H21	64551.19	47952.86	2401	110	Sonic
L3H23	64459.32	48008.64	2389	110	Sonic
L3H25	64351.05	48057.85	2376	110	Sonic
L3.5H0	65371.9	47298.97	2390.58	144	Sonic
L3.5H2	65280.64	47339.78	2404.38	124	Sonic
L3.5H2E	65463.11	47258.2	2388.72	144	Sonic
L3.5H4	65189.02	47380.85	2411.47	104	Sonic

Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
L3.5H4E	65554.67	47217.47	2396.89	110	Sonic
L3.5H6	65097.84	47421.28	2413.4	104	Sonic
L3.5H6E	65646.03	47176.96	2416.38	110	Sonic
L3.5H8	65006.15	47462.19	2412.41	104	Sonic
L3.5H8E	65737.5	47135.96	2419.72	110	Sonic
L3.5H10E(B)	65837.1	47131.8	2403	57	Sonic
L4H19	64456.07	47341.64	2441.85	115	Sonic
L4H21	64355.78	47369.47	2443.17	130	Sonic
L4H23	64261.56	47407.5	2440.31	110	Sonic
L4H25	64160.77	47445.19	2433.77	100	Sonic
L4H27	64057.36	47469.57	2428.34	105	Sonic
L4H29	63985.15	47528.49	2422.38	110	Sonic
L4.6H11	64612.48	46971.32	2462	75	Sonic
L7AH2	61050.28	47951.93	2386.91	104	Sonic
L7AH4	61085.02	48045.98	2383.54	104	Sonic
L7AH6	61094.64	48115.68	2381.57	94	Sonic
L7AH8	61118.85	48222.37	2386.45	94	Sonic
Ln12H1	63823.76	43328.29	2850	64	Sonic
Ln12H2	63934.91	43347.11	2831	64	Sonic
Ln12H3	63884.08	43374.21	2836	54	Sonic
Ln12H4	63798.23	43376.46	2882	42	Sonic
Ln12.5H2	63843.62	43131.46	2865	50	Sonic
LnAWP-0H0	63951.22	44452.05	2870.48	94	Sonic
LnAWP-0H2	63903	44363.9	2878.58	75	Sonic
LnAWP-0H4	63867.59	44283	2884.7	74	Sonic
LnAWP-0H14	63727.93	43791.42	2901.97	50	Sonic
LnAWP-0H16	63722.61	43702.17	2897.61	74	Sonic
Pit1	65745.81	47898.58	2480.36	30	Sonic
Pit2	65809.7	47938.1	2465.87	30	Sonic
Pit3	65700.3	47886.37	2484.31	75	Sonic
Pit4	65633.31	47971.93	2489.69	150	Sonic
Total				**6,253.7**	
2017 Drill Holes					
L2H12E	66380.37	46764.19	2356.36	61	Sonic
L2H13E	66418.95	47629.63	2354.86	85	Sonic
L2H14E	66466.32	47609.94	2352.23	70	Sonic
L2H15.5E	66543.93	47583.98	2348.36	63	Sonic
L2H18E	66668.35	47538.15	2346.13	79	Sonic
L2H20E	66749.31	47476.44	2353.9	59	Sonic
L2.25H13E	66336.37	47555.42	2360.96	71	Sonic
L2.25H15E	66412.47	47508.98	2359.19	65	Sonic
L2.25H19E	66585.71	47417.91	2359.74	61	Sonic
L2.25H21E	66712.4	47368.58	2370.7	73	Sonic
L2.5H14E	66333.68	47420.35	2368.64	95	Sonic
L2.5H17E	66456.09	47364.05	2367.98	90	Sonic

Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
L2.5H18E	66517.94	47338.25	2371.88	70	Sonic
L2.5H19E	66548.17	47294.5	2377.82	72	Sonic
L2.5H20E	66605.02	47309.46	2376.77	76	Sonic
L2.75H15E	66299.26	47343.56	2375.2	92	Sonic
L2.75H16E	66313.41	47316.69	2376.21	82	Sonic
L2.75H18E	66418.64	47258.47	2381.2	68	Sonic
L2.75H20E	66523.15	47247.67	2384.94	71	Sonic
L3H18E	66246.64	47218.46	2388.39	74	Sonic
L4H4E	65528.5	46939.44	2451.18	82	Sonic
L4H7E	65651.74	46912.46	2448.56	46	Sonic
L4H9E	65729.39	46899.38	2443.64	46	Sonic
L4.6H2E	65239.14	46754.02	2449.21	127	Sonic
L4.6H5E	65351.12	46708.69	2473.2	108	Sonic
L5H1E	65116.78	46586.79	2458.17	65	Sonic
L5.25H0.5	64963.93	46492.94	2470.93	100	Sonic
L5.5H1	64933.02	46379.46	2483.76	130	Sonic
L5.5H1E	65013.57	46363.06	2474.05	91	Sonic
L5.5H5E	65169.8	46269.08	2514.4	74	Sonic
L5.75H0.5	64917.55	46242.22	2494.59	130	Sonic
L6H0.5	64905.48	46154.69	2500.62	120	Sonic
L6H2E	65027.26	46063.22	2529.86	210	Sonic
L6H4.5	64735.34	46207.85	2555.68	85	Sonic
L6H5.5	64672.07	46219.27	2568.93	75	Sonic
L6.25H2.5E	64995.61	45954.61	2544.46	148	Sonic
L6.25H5	64715.52	46111.87	2568.31	105	Sonic
L6.25H6	64656.07	46093.01	2581.36	107	Sonic
L6.5H2E	65009.2	45848.14	2561.61	146	Sonic
L6.5H6.5	64623.64	45999.38	2596.82	83	Sonic
L6.75H3E	65024.69	45691.9	2579.89	145	Sonic
L6.75H6	64610.2	45850.05	2615.78	89	Sonic
L7H3.5E	65112.95	45499.59	2601.64	155	Sonic
L7H6.5	64665.16	45738.21	2613.55	84	Sonic
L7H7.5	64613.27	45757.16	2625.89	97	Sonic
L7.15H7	64646.65	45682.62	2630.18	104	Sonic
L7.25H2	64904.92	45448.08	2610.3	147	Sonic
L7.25H8	64628.49	45608.18	2648.39	100	Sonic
L7.5H0.5	64980.46	45273.2	2605.71	171	Sonic
L7.5H5.5	64738.28	45348.24	2663.55	117	Sonic
L7.5H7.5	64644.25	45442.33	2664.6	85	Sonic
L7.75H0	64936.22	45145.09	2624.67	168	Sonic
L7.75H4B	64809.18	45193.59	2655.28	90	Sonic
L7.75H6	64707.68	45264.27	2670.9	90	Sonic
L7.75H8	64603.07	45303.12	2684.74	90	Sonic
L8H9	64543.43	45201.39	2702.07	108	Sonic
L8.3H2	64853.15	44886.29	2653.54	141	Sonic

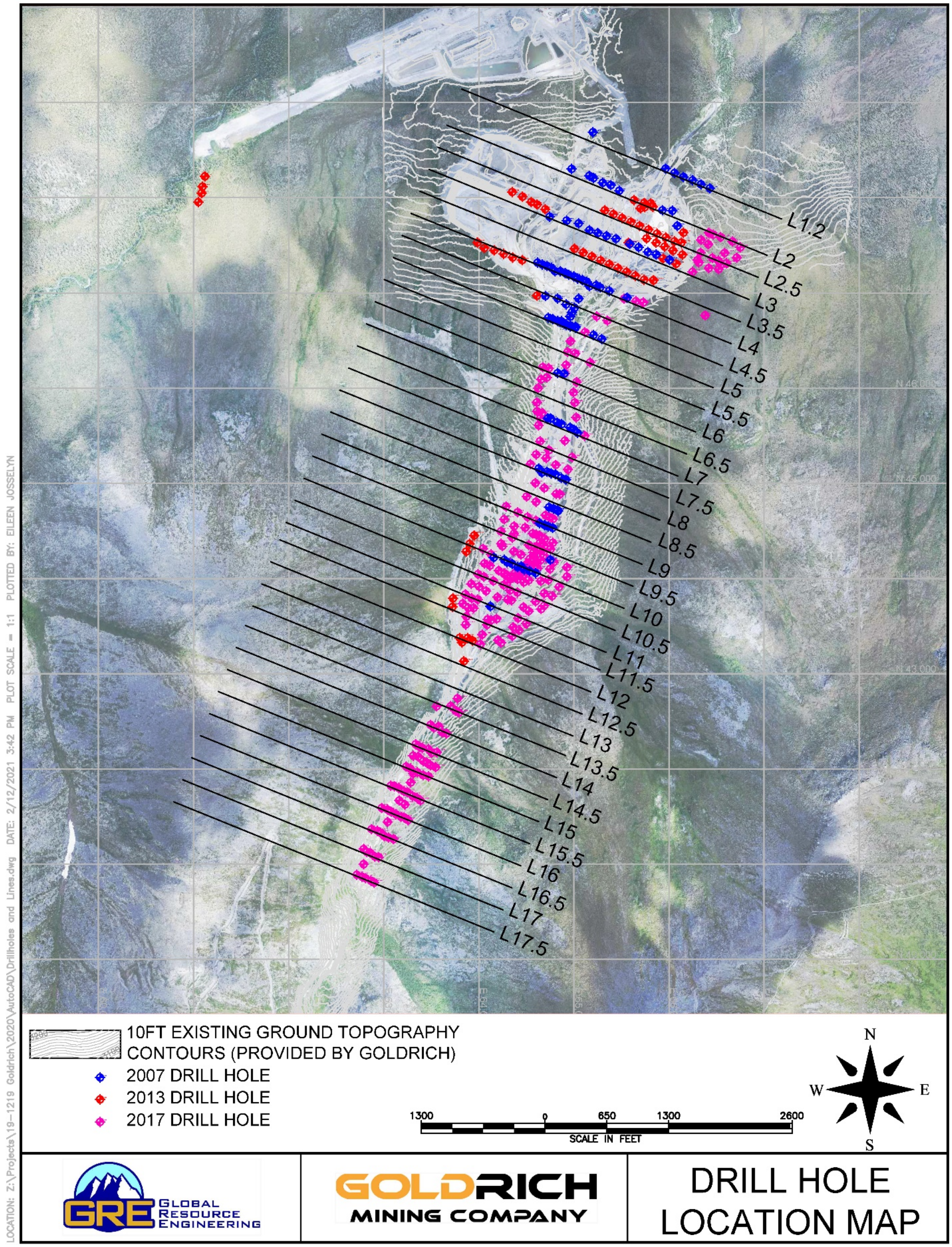
Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
L8.3H3	64814.44	44887.54	2665.55	70	Sonic
L8.3H6	64688.86	44949.56	2701.44	70	Sonic
L8.3H8	64594.47	45003.4	2713.88	110	Sonic
L8.6H6	64601.77	44805.43	2720.6	70	Sonic
L8.6H8	64502.44	44861.08	2744.32	100	Sonic
L8.8H1.5	64780.82	44626.37	2673.72	93	Sonic
L8.8H2.5	64740.24	44647.07	2672.64	80	Sonic
L8.8H3.5	64702.1	44663.38	2672.31	60	Sonic
L9H5	64601.11	44607.55	2695.47	92	Sonic
L9H7	64511.56	44646.29	2744.09	70	Sonic
L9H9	64408.73	44696.34	2770.83	100	Sonic
L9H12	64286.75	44756.54	2805.71	102	Sonic
L9.25H1	64778.3	44375.9	2681.69	64	Sonic
L9.25H3	64691.89	44415.82	2685.53	84	Sonic
L9.25H4	64637.77	44451.22	2679	75	Sonic
L9.25H5	64604.28	44460.51	2678.84	70	Sonic
L9.25H6	64567.3	44473.12	2694.32	90	Sonic
L9.25H8	64487.7	44534.57	2757.61	75	Sonic
L9.25H10	64371.15	44549.92	2782.74	110	Sonic
L9.25H12.5	64266.26	44621.41	2809.65	110	Sonic
L9.4H2	64753.88	44307.9	2685.76	42	Sonic
L9.4H4.5	64687.89	44342.7	2689.04	70	Sonic
L9.4H5.5	64642.51	44372.16	2672.67	51	Sonic
L9.5H3	64685.63	44285.53	2679.53	52	Sonic
L9.5H4	64638.21	44296.86	2694.91	102	Sonic
L9.5H4.5	64613.6	44315.47	2678.71	50	Sonic
L9.5H5	64584.33	44326.32	2695.83	30	Sonic
L9.5H6.5	64525.12	44353.69	2680.45	56	Sonic
L9.5H9.5	64352.39	44430.31	2793.08	59	Sonic
L9.5H9.5B	64369.22	44470.45	2786.48	163	Sonic
L9.5H13	64232.99	44495.35	2817.55	130	Sonic
L9.8H2.5	64539.33	44124.7	2699.84	30	Sonic
L9.9H0	64789.83	44172.02	2702.89	9	Sonic
L9.9H2	64712.46	44214.11	2692.59	43	Sonic
L9.9H3E	64927.15	44117.31	2777.36	47	Sonic
L9.9H4	64617.08	44260.82	2698.33	76	Sonic
L9.9H5	64575.11	44276.4	2690.06	57	Sonic
L9.95H1	64649.95	44193.95	2690.65	46	Sonic
L9.95H1E	64748.02	44100.09	2717.13	30	Sonic
L9.95H2	64606.71	44173.76	2683.33	50	Sonic
L9.95H2.5	64581.52	44185.67	2694.82	45	Sonic
L9.95H3.5	64535.52	44206.63	2700	52	Sonic
L9.95H5E	64927.24	44019.49	2793.37	60	Sonic
L9.95H5.5	64443.87	44250.86	2735.37	70	Sonic
L9.95H7.5	64368.49	44284.71	2741.96	97	Sonic

Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
L9.95H10.5	64248.06	44340.26	2812.66	119	Sonic
L9.95H11	64235.29	44366.01	2813.19	141	Sonic
L9.97H1	64617.51	44108.96	2699.11	53	Sonic
L9.98H3.5	64488.49	44140.49	2699.87	57	Sonic
L10H1E	64651.43	43972.74	2748.39	85	Sonic
L10H2E	64731.48	43984.77	2756.76	55	Sonic
L10H5E	64843.95	43912.81	2798.95	52	Sonic
L10H6.5	64329.69	44167.01	2738.16	70	Sonic
L10H11	64128.38	44253.29	2835.37	126	Sonic
L10H13	64043.26	44313.27	2849.74	93	Sonic
L10.15H0.5	64525.06	43981.91	2707.15	51	Sonic
L10.15H1.5	64462.08	44034.68	2712.43	50	Sonic
L10.15H2.5	64430.91	44032.34	2713.22	30	Sonic
L10.15H3.5	64380.27	44056.83	2706.89	41	Sonic
L10.25H0.5	64473.69	43961.3	2704.56	33	Sonic
L10.25H2	64406.44	43992.28	2715.09	30	Sonic
L10.25H3	64360.43	44011.26	2723.03	53	Sonic
L10.25H4	64303.88	44029.33	2696.72	24	Sonic
L10.25H4.0E	64687.24	43843.78	2786.42	67	Sonic
L10.25H5A	64267.18	44054.3	2747.67	45	Sonic
L10.25H5B	64264.26	44049.45	2747.67	73	Sonic
L10.25H6E	64770.64	43815.79	2803.71	54	Sonic
L10.25H6.5	64209.46	44082.94	2760.73	103	Sonic
L10.25H8	64117.3	44135.21	2838.81	104	Sonic
L10.25H9.5	64060.2	44158.31	2848.98	110	Sonic
L10.4H2	64299.45	43958.94	2718.34	30	Sonic
L10.5H0	64327.94	43876.61	2707.87	21	Sonic
L10.5H2	64259.76	43932.99	2705.12	21	Sonic
L10.5H2E	64405.28	43854.38	2711.48	27	Sonic
L10.5H3.5	64153.19	43973.42	2777.56	75	Sonic
L10.5H5.5	64053.81	44018.78	2837.17	119	Sonic
L10.5H7	63991.7	44045.55	2854.04	111	Sonic
L10.5H7.0E	64596.76	43759.64	2800.3	61	Sonic
L10.5H8E	64689.15	43716.86	2814.27	46	Sonic
L10.7H0.5E	64304.25	43779.12	2715.88	21	Sonic
L10.7H1.5E	64342.15	43770.13	2715.95	22	Sonic
L10.75H1.5	64196.06	43803.4	2713.56	29	Sonic
L10.75H4	64075.56	43868.49	2797.64	56	Sonic
L10.75H4.5E	64469.47	43698.79	2806.1	91	Sonic
L10.75H5.5	64002.62	43915.3	2838.16	90	Sonic
L10.75H5.5E	64524.54	43657.71	2813.98	69	Sonic
L10.75H6.5	63921.28	43943.69	2860.96	75	Sonic
L10.75H7.5E	64655.93	43679.54	2816.93	49	Sonic
L11H5	63989.87	43779.35	2811.42	71	Sonic
L11H5.5E	64411.43	43584	2823.29	93	Sonic

Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
L11H6.5	63924.63	43792.13	2840.88	63	Sonic
L11H6.5E	64498.3	43526.39	2843.27	59	Sonic
L11H7.5	63859.15	43833.41	2872.77	80	Sonic
L11.25H1E	64187.15	43576.19	2744.07	25	Sonic
L11.25H4.5	63944.79	43690.3	2833.23	66	Sonic
L11.25H5.0E	64361.6	43488.97	2836.45	89	Sonic
L11.25H5.5	63889.42	43691.12	2853.28	80	Sonic
L11.25H6E	64420.64	43459.39	2849.57	60	Sonic
L11.25H7	63831.04	43736.24	2878.81	106	Sonic
L11.3H0	64129.77	43562.49	2758.6	30	Sonic
L11.3H1	64093.7	43571.06	2753.49	31	Sonic
L11.5H0	64085.54	43513.39	2764.5	17	Sonic
L11.5H4.5	63868.99	43601.25	2854.82	55	Sonic
L11.5H5.5E	64352.73	43396.92	2856.69	57	Sonic
L11.75H0	64035.34	43411.18	2782.61	19	Sonic
L11.75H3.5	63861.68	43516.16	2858.73	62	Sonic
L11.75H4.5E	64202.31	43351.15	2853.67	80	Sonic
L11.75H5.5E	64234.82	43325.02	2863.94	69	Sonic
L12H1E	64016.81	43309.41	2795.18	14	Sonic
L13.25H0	63776.29	42740.62	2890.42	31	Sonic
L13.5H0	63756.15	42609.79	2904.27	24	Sonic
L13.5H1	63704.87	42659.53	2907.22	44	Sonic
L13.5H1E	63790.56	42591.86	2917.59	50	Sonic
L13.5H2	63557.3	42658.22	2914.73	33	Sonic
L14H0	63614.74	42384.96	2927.07	29	Sonic
L14H1	63578.96	42445.88	2920.67	29	Sonic
L14H1E	63660.21	42372.13	2938.45	35	Sonic
L14H2	63529.14	42432.56	2933.2	42	Sonic
L14H3	63493.05	42462.38	2945.7	33	Sonic
L14.5H0	63470.86	42207.6	2945.34	23	Sonic
L14.5H1	63404.84	42220.88	2946.72	30	Sonic
L14.5H1E	63503.44	42167.93	2954.33	24	Sonic
L14.5H2	63371.34	42236.66	2957.48	37	Sonic
L14.5H2E	63558.97	42173.28	2963.45	26	Sonic
L14.5H3	63335.97	42258.51	2969.75	15	Sonic
L14.75H0	63414.58	42082.93	2955.64	18	Sonic
L14.75H1	63362.88	42127.49	2952.49	27	Sonic
L14.75H1E	63456.99	42062.26	2963.68	28	Sonic
L14.75H2	63309.56	42138.05	2975.39	43	Sonic
L14.75H2E	63494.22	42037.71	2971.62	17	Sonic
L14.75H3E	63545.47	42017.53	2984.38	15	Sonic
L15H0	63354.2	41963.96	2964.21	18	Sonic
L15H1	63309.8	41951.94	2969.19	27	Sonic
L15H1E	63413.88	41933.97	2971.95	20	Sonic
L15H2	63281.3	42002	2975.46	35	Sonic

Hole #	Easting (feet)	Northing (feet)	Elevation (feet)	Total Depth (feet)	Hole Type
L15H2E	63471.64	41922.11	2982.81	34	Sonic
L15H3	63236.05	42024.2	2990.03	40	Sonic
L15.25H1	63284.2	41857.34	2977.17	16	Sonic
L15.5H0	63304.99	41713.83	2991.21	19	Sonic
L15.5H1	63250.52	41753.81	2985.79	26	Sonic
L15.5H1E	63351.25	41693.65	3002.3	23	Sonic
L15.5H2	63211.62	41764.83	2993.64	36	Sonic
L15.5H2E	63386.5	41667.58	3008.53	25	Sonic
L15.5H3	63160.1	41797.47	2997.7	36	Sonic
L15.5H4	63111.83	41818.06	3008.92	37	Sonic
L15.5H5	63074.82	41823.06	3020.8	30	Sonic
L15.75H0	63223.1	41625.12	2995.54	13	Sonic
L15.75H1	63163.42	41668.1	3005.28	30	Sonic
L16H0	63128.22	41524.67	3006.69	10	Sonic
L16H1	63079.01	41549.33	3011.55	19	Sonic
L16H1E	63180.69	41496.54	3017.26	15	Sonic
L16H2	63041.15	41572.13	3026.48	24	Sonic
L16H2E	63214.83	41491.43	3021.23	18	Sonic
L16H3	62979.01	41589.67	3039.83	28	Sonic
L16H3E	63256.89	41462.83	3029.17	12	Sonic
L16.25H1	63040.42	41443.12	3014.76	12	Sonic
L16.5H0	63049.39	41279.05	3026.97	11	Sonic
L16.5H1	63005.54	41295.62	3033.2	17	Sonic
L16.5H1E	63103.6	41261.98	3044.55	18	Sonic
L16.5H2	62970.43	41327.76	3040.12	26	Sonic
L16.5H3	62933.59	41350.09	3047.31	28	Sonic
L16.5H4	62871.66	41367.24	3059.22	27	Sonic
L17H0	62906.38	41079.37	3045.9	13	Sonic
L17H1	62859.22	41116.89	3055.68	24	Sonic
L17H1E	62948.7	41072.78	3049.7	17	Sonic
L17H2	62823.16	41127.73	3063.85	25	Sonic
L17H3	62788.63	41158.16	3073.65	18	Sonic
L17.25H0	62788.79	40998.4	3059.68	13	Sonic
L17.5H0	62717.42	40887.18	3075.66	18	Sonic
L17.5H1E	62772.97	40866.72	3092.45	17	Sonic
L17.5H2E	62809.81	40851.3	3094.78	22	Sonic
L17.5H3E	62865.89	40825.24	3101.31	29	Sonic
L17.5H4E	62899.64	40806.37	3108.96	18	Sonic
Total				**14,303.0**	

Figure 10-2: Drill Holes Location Map



11.0 Sample Preparation, Analyses and Security

Pan and trench sampling were performed as early-phase exploration in order to collect sufficient information on which to base future drilling. Drilling was performed in 2007, 2013, and 2017, and the collective results were used to support the mineral resource estimate presented in Section 14 of this report.

11.1 Pan Sampling

Reconnaissance pan samples of placer material collected from old works and newer excavations (grab and channel samples) were measured by counts of heaped pans and converted to equivalent cubic units. Weighable amounts of gold that were found in the pan samples were saved in glass vials for later analysis.

11.2 Trench Sampling

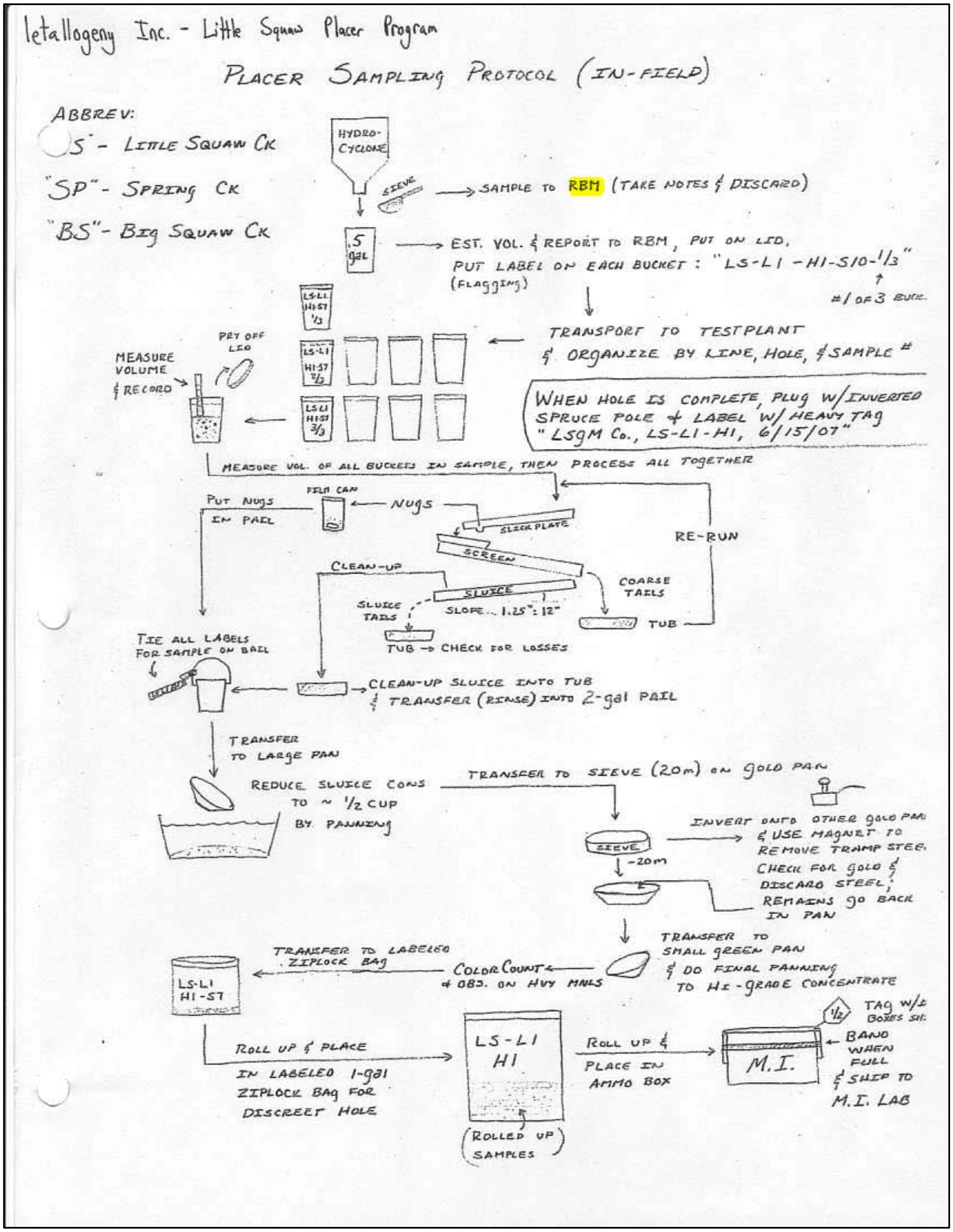
Trenches were sampled by placing overburden and waste material on one side of the trench and placing sample intervals in segregated stockpiles on the other side of the trench. Targeted placer sections were sampled on two-foot vertical intervals. Samples were collected from the stockpiles by cutting a trench through the stockpile to expose the interior, then shoveling small amounts of gravel from the cut stockpile to generally represent all the material in the interval. Trench samples measuring approximately one bank cubic foot were estimated by filling two 5-gallon buckets to level and panning; the concentrated samples were saved in glass vials for later analysis.

11.3 Drill Sampling

An independent contractor, Metallogeny, was employed to perform the 2007 drill hole sampling and processing. Conventional sampling methods were used to collect samples from all drill holes. An air-rotary, reverse-circulation drill (Alsinco A-80) driving 5 7/8-inch, 6-inch, and 6 1/8-inch tri-cone bits was used with an air compressor (Sullair) having a capacity of 750 cubic feet per minute and 350 pounds per square inch. Samples were collected from the surface to bedrock every five feet in 2007 and 2013, and every two feet in 2017. A sample of bedrock chips from the last drill interval was assigned a separate sample number and submitted for gold assay plus a multi-element analysis. A split of the chips of each bedrock intercept was retained in standard chip trays for a permanent bedrock record. Metallogeny adhered to a formal, in-field sampling protocol during all sample collection (Figure 11-1). The NyacAU program also adhered to an in-field sampling protocol which was carried out by in-house geologists.

Drill samples were collected at the underflow of a hydrocyclone in 5-gallon buckets placed in a small tub to catch the overflow of turbid water. Slimes recovered in the tub were added back to the sample buckets. Typically, two to three buckets of placer material were collected per sample interval. Lids were snapped on the buckets, and labels tied to the bails indicated deposit, hole number, sample number, and number of buckets per sample (i.e., “LS-L4-H3-S22, 1 of 3”). Notes on stratigraphy were made as drilling progressed down-hole. Initially, holes were drilled without water, which caused a problem with dust. To alleviate the health hazard and erratic sample volumes caused by drilling dry, water was injected. Adding water improved recovery and mitigated the excessive dust. The placer sample buckets were transported to a central processing site with a Bombardier tracked vehicle and trailer. Samples were unloaded and sorted according to hole, and an inventory was made to account for all sample material. Wet samples settled

Figure 11-1: In-field Placer Sampling Protocol for Drilling



and clarified overnight so that a more accurate measurement of the volume of solids could be made. The volume of each sample was measured with a measuring stick, recorded in the inventory notebook, and re-entered into the Field Panning Log.

11.3.1 Process

Interval samples were processed continuously for an entire hole. Sample reduction required a field hydraulic concentrator: a slickplate, a vibrating screen with ¼-inch openings, and a riffled sluice box. Placer samples were slurried onto the slickplate, then screened. Screen vibration was activated using a pelton wheel turning an off-balance shaft. A manifold provided the high-pressure water jets to the feed, disaggregating (slurrying) the sample, and to the pelton wheel. The sluice was cleaned by removing the heavy expanded metal riffle set and sliding the un-backed Nomad carpeting into a plastic tub. The entire test plant was washed to mitigate contamination and to remove particles trapped in the weaving. It was visually inspected for gold nuggets, then the sluice was re-assembled for the next sample. The sluice concentrate was poured into a bucket, and the original labels from the drill sample were tied to the bucket's bail. The sluice concentrate buckets were organized in preparation for panning. Sluice tailings were occasionally checked by collecting a full gold pan of tailing sands, just below the outfall, then panned to look for gold colors.

11.3.2 Panning

Sluice concentrates were panned over 18-gallon tubs using large, plastic gold pans. Standard panning techniques, including double-panning, reduced the sample to a high-grade pan concentrate. A plunge-magnet removed the magnetic material, such as minor amounts of magnetite and pyrrhotite and occasionally abundant amounts of tramp iron (shards and threads of drill tooling). The magnetic concentrates were described in the Field Panning Log by composition and relative abundance, checked for any gold possibly entrained between magnetic particles, then discarded. Further panning reduced the concentrate to reveal visible gold. A count of all visible gold colors was made according to a visual classification scheme based on estimated gold particle weight. The color count and estimated weight were recorded in the Field Panning Log, along with a description and relative abundance of the other heavy minerals. The placer gold was not removed from the pan concentrate. The pan concentrate was rinsed into a labelled plastic bag with a zip-lock closure. Most of the water was decanted, the bags were rolled to squeeze out the air, then sealed and placed into a larger, heavy-duty, plastic Ziplock bag. When the entire hole had been panned and the individual samples accumulated, they were sealed together into yet another bag and placed in a U.S. Army ammunition steel can for secure transport.

11.3.3 Storage and Transport Security

Placer sample concentrates, stored in U.S. Army ammunition cans, were placed into a heavy-duty aluminum strong box in the drill geologist's office. On a purposely semi-regular basis, the pan concentrates were re-inventoried by drill line and by hole, then re-packed into the ammunition cans with copies of the corresponding Field Drill Logs and Field Panning Logs. The cans were sealed with a heavy-duty steel strap and sent to Fairbanks via chartered air taxi. A representative from the contractor (Metallogeny, Inc.) met the aircraft at touchdown in Fairbanks or soon after if the samples were in the possession of supervisory company personnel. The sample containers were then transported to the laboratory and stored for analysis.

11.4 Laboratory Analysis

Placer concentrates received in the Fairbanks laboratory were immediately entered into the laboratory's Sample Inventory Log to check for missing sample intervals. Samples were then organized and double-panned into a tub in which all pan tailings were accumulated (composited) for a discreet drill hole. A high-grade pan concentrate was produced, from which all visible gold was extracted using a pipette. The gold was transferred to a labelled glass vial for further analysis. The remaining pan concentrates were transferred to an aluminum weighing boat and dried over low temperature heat. When dry, the heavy minerals were described by relative abundance with the aid of a binocular microscope. The pan concentrate was weighed and placed in a labelled paper envelope for storage. The accumulated pan tailings for each hole were dried and saved in a labelled sealed plastic bag.

The sample vials containing the placer gold particles were organized and analyzed by transferring the gold into a finishing pan for a final concentration and rinse under ethanol. In skilled hands, a nearly clean separation of gold from gangue can be made, and the resultant fractions were dried in the finishing pan over low-temperature heat. The gold particles were picked out with a razor blade (or toothpick) and placed on a formed piece of gridded paper in groups according to particle mass. Any remaining gangue minerals were added to the accumulated pan tailings for the drill hole. Gold particles were classified visually into five weight categories in order to conduct a color count. The mass of each gold color was estimated and grouped according to the color table:

- Nuggets -- >150 milligrams (mg)
- C -- 5 to 150 mg
- B -- 2 to 5 mg
- A -- 1 to 2 mg
- f -- <1 mg ("flyspeck")

A digital image of each sample with visible gold was captured, edited, and saved on electronic media. The sample was then weighed with an Ohaus™ digital analytical balance, sensitive to 0.1 mg. Nuggets were weighed and recorded individually. The sample was recombined with the nuggets and placed in a clean, labelled glass 2-dram vial. All the sample vials for a hole were bundled in a labelled cloth sample bag. The gold particles (vials), pan concentrates (paper envelopes), and pan tailings (plastic bag) were then placed in a custom, high strength, labelled cardboard box for storage. These smaller boxes were then placed into larger, heavy-duty labelled cardboard boxes, organized by drill line.

Significant measures were taken to mitigate sample contamination, whether by carrying of very fine grains of gold and other heavy minerals from one sample to the next or by exotic material introduced from the laboratory environment. All quantitative and descriptive data were entered onto the Weigh-Up Log. A unique label was created at every stage of sample transfer so that the integrity of sample identity was secure.

12.0 Data Quality and Verification

GRE QPs Terre Lane and Hamid Samari were accompanied by Dick Walters, a consulting economic geologist, during an on-site inspection of the project on September 4 and 5, 2019. In addition to visiting the Chandalar Mine placer area, the party toured all existing facilities and observed all equipment remaining from previous phases of exploration and mining activities.

12.1 Database Validation

12.1.1 Collar Coordinate Validation

The locations of drill holes in 2007 and 2013 were originally collected in NAD83 grid while that 2017 in WGS84 latitude, longitude, and elevation.

Both phases of surveying used differential GPS. Year 2007 used Magellan Mobile Mapper Pro global positioning system (GPS) instrument, consisting of a stationary base unit and a rover unit equipped with external antennas, with post-processed location accuracy of 70 cm for horizontal and vertical datum. Year 2017 used high-precision GPS of unknown model with accuracy <1 cm to collect the location data.

The 2007 and 2013 drill hole locations were converted from NAD83 to a local mine grid by site and presented in autoCAD format; the 2017 drill hole locations were converted from WGS84 lat/long to the local mine grid by site, using the Carlson data collection and drawing software and were presented in excel format.

Drill hole collar locations and orientations are typically ground-truthed in the field using a hand-held GPS unit and a Brunton compass, but this was not possible at the Chandalar project because drill hole collars have been obliterated by mining activity. In lieu of checking the individual collar locations, the QPs checked the approximate locations of the drilling lines at Chandalar Mine from line 1.2 in the north to line 10 in the south (Photo 12-1).

Photo 12-1: Field Evaluation of Part of Chandalar Mine

The QPs recommend that future drill holes, especially in lode deposits, be surveyed using a differential GPS in UTM WGS84 coordinates and that all previously completed drill holes be converted to UTM WGS84 coordinates. These coordinates should then be compared to the digital topography in areas where LIDAR data is available. In areas where only topography data was generated during the magnetic ZTEM survey, the differential GPS would likely provide a more accurate representation of the terrain.

12.1.2 Down-Hole Survey Validation

Down-hole survey data are typically validated by identifying any significant discrepancies between sequential dip and azimuth readings; however, there is no mention or indication of this process in the data and documents provided by Goldrich. Down-hole surveys were not completed on the holes drilled at Chandalar Mine because all were drilled vertically and without casing. Given the shallow depth of the holes in general, the lack of down-hole survey is considered unimportant.

12.1.3 Check Gold Grade

In 2007, the mass of gold and the sample volume was recorded, and the gold grade was calculated with a factor of volume recovery as follows:

$$\frac{\text{mass of gold (mg)}}{volume\ recovery\ (\%)\ x\ 31103.5}\ x\ \frac{27}{5\ x\ 0.1888}\ x\ \frac{870}{1000},$$

which gave fine gold in ounces per bank cubic yard (oz/bcy). The 870/1000 gold purity was inverted to give the raw gold grade for resource estimation.

In 2013 and 2017, the mass of gold and either sample height in bucket or sample volume was recorded on the field sheet. The raw gold grade was then calculated from the formula:

$$\frac{\text{mass of gold (mg)}}{31103.5}\ /\ sample\ volume\ (yd3)$$

Conversion of sample volume from the sample height in the bucket for some of the samples was not shown.

This formula was used to calculate the final gold grade and was shown on the excel spreadsheet for 2007 but not for 2013 and 2017 samples. The 2007 data appears to be of good quality, in terms of calculation of gold grade. For 2013 and 2017, the conversion from bucket height to sample volume was calculated, but a small amount of missing data on both bucket height and sample volume makes it difficult to verify all the calculations.

The QPs checked and evaluated all gold grade (oz/bcy) results from drill samples from the Little Squaw placer mine through 2017, as recorded in the Goldrich data base. The sample results are summarized in Table 12-1.

Table 12-1: Range of Gold Grade in Drill Samples from Chandalar Mine

Range of Gold Grade (oz/bcy)	Number of Sample Intervals
> 2.3	0
2.3 to 1	9
1 to 0.5	19
0.5 to 0.1	258
0.1 to 0.05	242
0.05 to 0.01	1,017
0.01 to 0	1,716
0	7,505
Total Number of Sample Intervals	**10,756**

In all drilling campaigns, no blank, standard, or duplicate samples were inserted into the sample stream which would be typical for hard rock lode deposits. This is common practice for placer gold deposits where the coarse free gold requires large samples to be representative. The entire sample is processed to a pan concentrate revealing the gold specs, flakes, and nuggets which are removed and weighed. The purpose of a blank is to ensure there is not contamination in the laboratory resulting in grade occurring where there is none. Inserting a blank (gravel known to not contain placer gold) would tell nothing. Standards (samples with known grade) are inserted into a sample stream to ensure the accuracy of the analysis. Creating a standard for a placer deposit might involve taking gravel known to be barren and adding a few flakes of gold of known mass to each sample. This might give one information about panning efficiency but tells us nothing about the accuracy of the analysis. Dividing a sample in two, to create a duplicate would create other problems. A small nugget might report to one sample and nothing reports to the other. Coarse free gold also presents problems due to gravity segregation even at a sample prep stage.

The sample grade can only be verified by drill holes in close proximity, like twin holes, and statistical analysis.

GRE's QPs collected five 5-gallon buckets from the lower gold bearing fluvial section of the Chandalar Mine deposit for panning at the site (Photo 12-2). Samples were selected from different parts of the deposit, longitudinally through the placer (

Figure 12-1), and were collected by hand shovel. The samples were panned at the site, and the resultant fine size portions were collected into vials as panned concentrates for laboratory testing (Photo 12-3).

Photo 12-2: QP Data Validation Sampling from the Chandalar Mine

Figure 12-1: GRE Sample Locations at Chandalar Mine

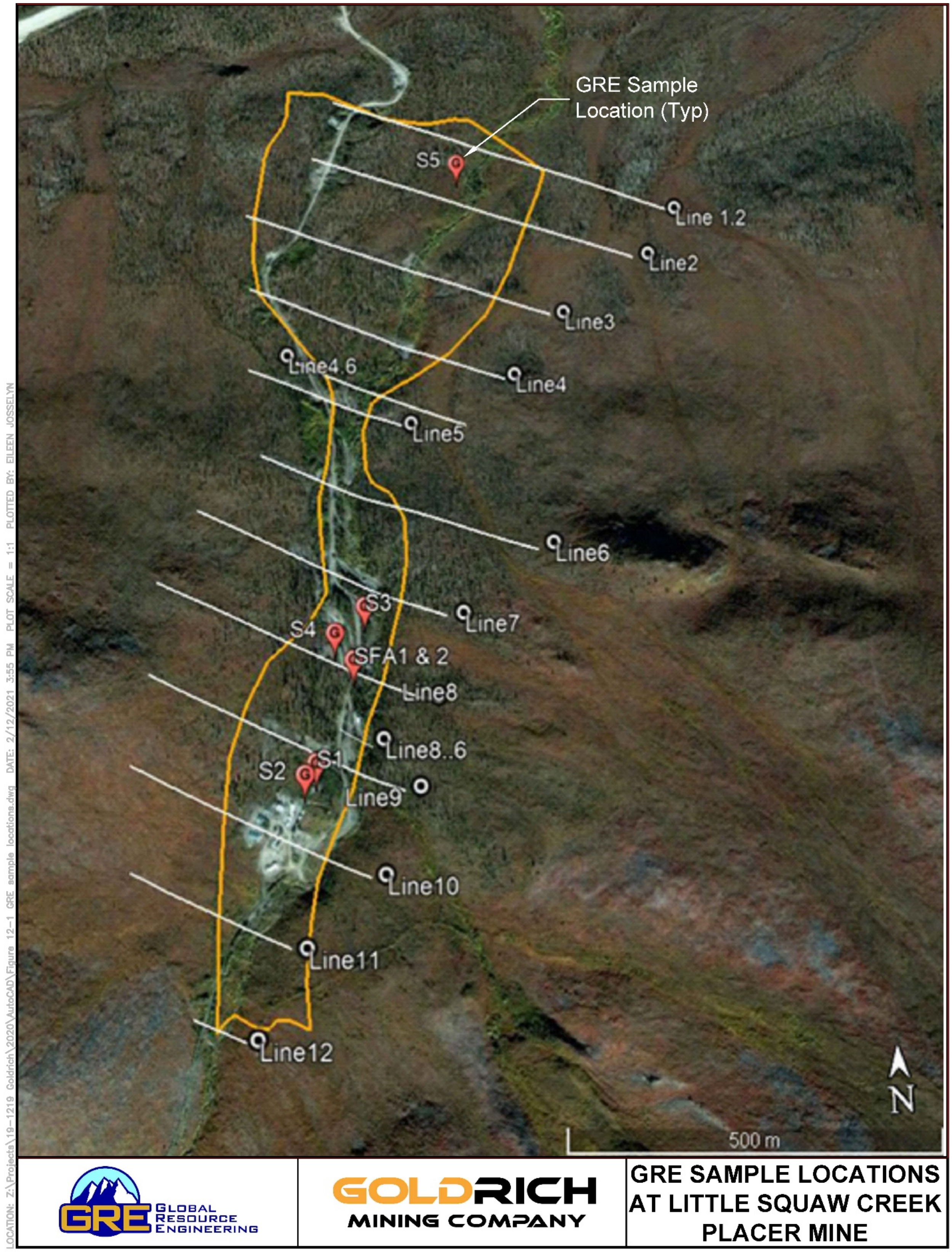


Photo 12-3: GRE Panning at the Chandalar Mine and Collecting the Fine Size in the Vial

The QPs also selected two chip rocks samples for assay analysis. One sample was taken from the contact of schist and greenstone and the other from the contact of schist with quartz veins. A total of seven samples, including the five panned samples and two chip samples, were packed and transported by the QPs to Resource Development Inc. (RDi) in Golden, Colorado, USA, for analysis (Table 12-2, Photo 12-4). Analytical results are summarized in Table 12-3.

Table 12-2: Check Samples Submitted to RDi Labs

Sample No.	GPS Location No.	Sample ID	Type of Sample	Vial No.	Analysis Requested	
					Weight of Au/Ag Bead	Fire Assay-Au/ICP
1	49	G-GRE001	Panned Concentrate	1	✓	
2	50	G-GRE002	Panned Concentrate	2	✓	
3	51	G-GRE003	Panned Concentrate	1	✓	
4	52	G-GRE004	Panned Concentrate	2	✓	
5	55	G-GRE005	Panned Concentrate	2	✓	
6	53	G-GRE-SFA-01	Chip surface sample			✓
7	53	G-GRE-SFA-02	Chip surface sample			✓

Photo 12-4: Sample Verification at GRE's Denver Office

Table 12-3: Summary Table of RDi Results

Sample No.	GPS Location No.	Sample ID	Type of Sample	Results of the Analysis	
				Weight of Gold Bead (mg)	Assay (ppm)
1	49	G-GRE001	Panned Concentrate	<0.01	
2	50	G-GRE002	Panned Concentrate	<0.01	
3	51	G-GRE003	Panned Concentrate	28.35	
4	52	G-GRE004	Panned Concentrate	<0.01	
5	55	G-GRE005	Panned Concentrate	<0.01	
6	53	G-GRE-SFA-01	Chip surface sample		0.007
7	53	G-GRE-SFA-02	Chip surface sample		0.007

To calculate gold grades, the QPs used both of the formulas previously applied through drilling campaigns in 2007, 2013, and 2017. Results for the five pan samples are presented in Table 12-4.

Table 12-4: Pure Gold Grade of Five Panned Concentrated Samples by QPs

Sample No.	Sample ID	Type of Sample	Weight of Gold Bead (mg)	Pure Gold Grade (oz/bcy)
1	G-GRE001	Panned Concentrate	<0.01	1.63E-05
2	G-GRE002	Panned Concentrate	<0.01	1.63E-05
3	G-GRE003	Panned Concentrate	28.35	0.046
4	G-GRE004	Panned Concentrate	<0.01	1.63E-05
5	G-GRE005	Panned Concentrate	<0.01	1.63E-05

GRE QPs collected five 5-gallon bucket samples from different horizons of the Chandalar Mine for panning and analysis. The samples were taken near drilling lines L9, L7, L8, L1.2, and L2 (see Figure 12-1). After panning and preparing concentrate samples, Ms. Lane sent the five pan concentrates to the RDi laboratory in Golden to be melted, which produced a gold bead that was then weighed.

Table 12-3 shows the weight of the gold bead for all five samples. Only one of the five samples had a measurable gold bead, and it weighed 28.35 mg. A small nugget (or large flake) was seen in that pan during panning (Photo 12-3). GRE applied the gold fineness for Chandalar gold (840 fine) and the bucket volume and determined the sample contained about 0.05 (oz/bcy) of gold (Table 12-4).

The other four panned concentrate samples were barren of gold. The lack of gold can be attributed to two things:

- Sampling within the placer pit was challenging. The pit walls are high, and the exposed gravel is located at current pit boundary, which is presumably the mineralized limit, there are many areas within the pit that have exposed glacial till on the bench, and the pay gravel is many feet below the current bench.
- The sampling theory and sampling practice described in detail by Pitard (1993) are problematic. Large samples are needed when the size of the sampled payable particles is large. Placer deposit gold has gold particles measured in tenths of an inch, where disseminated gold deposits have gold particles measured in microns. The small samples (5-gallon buckets) GRE took are too small to be

representative in a placer deposit, and taking larger samples was not feasible with the equipment available. When samples are too small in coarse gold, one gets many very low-grade results and an occasional high-grade result. Each of these results are not representative, the low grade is too low, and the high grade is too high. The real grade of the sampled material is likely an average all of the samples.

Ms. Lane also took two chip rock samples to help understand the nature of the lode mineralization (Figure 12-1 and Photo 12-4), and its relation to the placer gold. One sample was a quartz vein and schist, where the quartz vein was parallel to the schistosity, and the other sample was schist.

The assay results were negative and did not show gold in these two chip samples. The result from the schist sample confirmed that there is no gold associated with schists (see mineralization section). The other sample, quartz vein in schist, shows that all quartz veins are not gold-bearing quartz veins in the Little Squaw catchment. As mentioned in the load mineralization section of this report, at least two or three different types of the quartz veins in the Chandalar Mine catchment exist; only some of them are associated with gold.

Dr. Samari believes there is an excellent potential for lode deposits at Chandalar Gold Project property. An exploration program for lode deposit should consist of detailed mapping and geophysics studies to help identify the gold-bearing quartz veins and fault zones. This work should be followed by exploration drilling of the quartz vein and fault zones eventually over the entire Chandalar property.

12.2 Geological Data Verification and Interpretation

12.2.1 Geological Map Accuracy

Field observations made during the site visit generally confirm previous reports and maps of the geology of the project area. The Chandalar Mine, from drilling line 1.2 in the north to line 10 in the south, was inspected, and the QPs did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting of the Little Squaw placer deposit (Photo 12-5). Placer location, sedimentary horizons, and structural and morphologic features are all consistent with descriptions provided in previous project reports (Barker, et al., 2009; Mendham, et al., 2018).

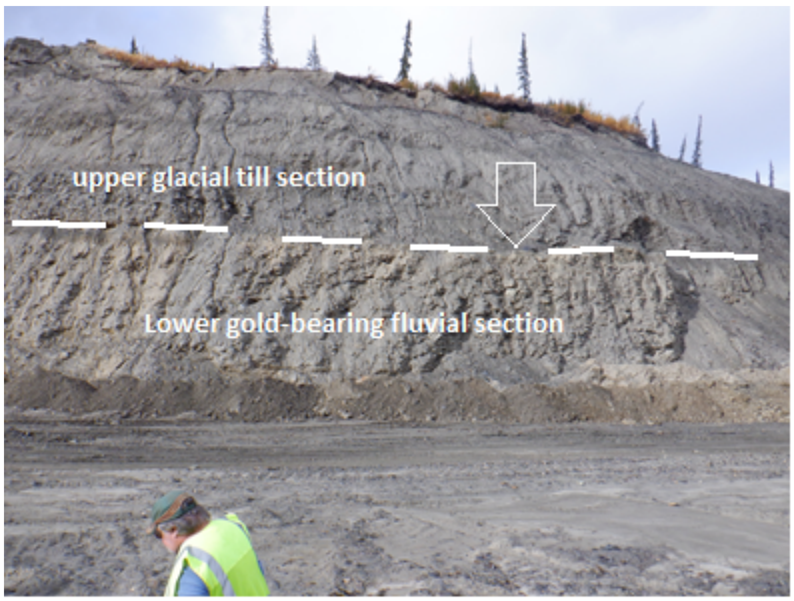
Photo 12-5: A general view of the Chandalar Mine (view to the southwest)

12.2.2 Geological Logging Accuracy

In addition to the Chandalar Mine placer area, the QPs visited dump areas to check waste material gold content by panning. Most of the material in the dump areas consist of the schists, though fragments of quartz and lesser greenstone were also observed.

While there are no intact drill samples to use for validation purposes, the Ms. Lane and Dr. Samari did check the different horizons of the placer deposit as described in previous technical reports (Barker, et al., 2009; Mendham, et al., 2018). As shown in Figure 12-2, two different horizons of the deposit are apparent: a barren upper glacial till section and a lower, gold-bearing fluvial section. In all cases, the field evidence supports previous descriptions of placer horizons in the Chandalar Mine deposit.

Figure 12-2: View of Two Horizons of Sediments (Permafrost) in the Chandalar Mine



Located between drilling lines of 7 and 8 (view to the west)

12.3 QP Opinion on Adequacy

Based on the results of the Ms. Lane and Dr. Samari's review of the sampling effort, verification of drilling lines and their profiles in the field, gold grade check sample analysis, the results of the QPs' on-site sampling and panning, and the results of both manual and mechanical database audit efforts, Ms. Lane considers the gold grade data contained in the project database to be reasonably accurate and suitable for use in estimating mineral resources.

Although some parts of the deposit were removed in the previous mining and operating periods of 2015, 2016, 2017, and 2018, all exploration data for the remaining un-mined portions of the placer are still useful and valid.

Ms. Lane finds the mining methods, operating and capital costs suitable for use in economic analysis. Dr. Harvey finds the recovery methods, process operating and capital costs suitable for use in economic analysis.

13.0 Mineral Processing and Metallurgical Testing

13.1 Metallurgy Summary

There is no conventional metallurgical test work associated with the Chandalar Mine. Typically, placer deposits are analyzed by drill hole, test pit, or trench sampling. The material from these samples is subjected to gravity gold recovery usually using a portable trommel and sluice system. The recovered concentrate is typically cleaned on a shaking table and the doré assayed. From this test work, a recovered grade is calculated and applied to the deposit, normally measured in oz/bcy. Figure 11-1 shows the In-Field Placer Sampling Protocol. Through geologic methods, this recovered grade is translated to the mining units. At this stage of the project, historic mining (see Section 6.0 History) provides greater detail on the processing methods and metallurgical performance than laboratory test work can.

The placer gold occurs within the pre-glacial fluvial, interglacial glaciofluvial, and post-glacial fluvial deposits. The gold is generally coarse, crystalline, and bright, indicating that it was transported only a short distance from its sources. Small nuggets are common, and there is little fine-grained gold (-80 mesh). Quartz inclusions and attachments are common on gold particles but make up only a few percent by volume.

The process method, gravity concentration, recovers gold particles that have been liberated from the host rock. The sampling method similarly recovers the liberated gold particles, thus gold recovery during operations is estimated to be 100 percent of the sampled grade. Historical refinery records show the gold from the Chandalar placer are 84% pure (840 fine); GRE applied the 84% purity factor in the economic analysis.

13.2 Historical Production

In 2009, Goldrich completed an alluvial gold mining test at Chandalar Mine. The pilot program involved a mining test that extracted approximately 594 “raw” ounces of placer gold, equivalent to about 488 ounces of fine gold. The extracted material averaged approximately 835 fine or 83.5% gold and 16.5% silver.

These results lead to the development of the project and the start of larger scale mining in the summer of 2010. The project has specific horizons within it that are up to 20 feet thick containing the richest gold grades. The mineralized material is about forty percent composed of gravel, cobbles, and boulders set in a sixty percent matrix of fine silt. It is not frozen below twelve to fifteen feet of depth.

The full potential of the initial project was not realized due to a shortage of working capital. At the end of the 2010 mining season, Goldrich had extracted 1,924 ounces of gold concentrate, from which approximately 1,509 ounces of fine gold and 259 ounces of fine silver were extracted. Since 2010, the Chandalar Mine has produced approximately 42,212 ounces of fine gold, as summarized in Table 13-1 (rounded).

The gold recovery methods employed for the documented mining above was through gravity means. The most recent operation employed a traditional wash plant consisting of a feed hopper with water sprays to move the material into the associated trommel. The rotating trommel was also equipped with water sprays, the oversize material was transported out the end of the trommel, and the undersize reported to a series of sluice boxes fitted with riffles and miner's moss. The placer materials collected in the sluice and were periodically recovered, further upgraded on a shaking table, dried, and sold to a refinery.

Table 13-1: 2013 to 2018 Gold Production at Chandalar Mine

Year	Ounces of Placer Gold	Ounces of Fine Gold	Tonnage Treated (bcy)	Recovered Grade (oz/bcy)
2013	937	694	?	?
2014	0	0	0	0
2015	4,697	3,853	104,620	0.0372
2016	10,203	8,226	329,050	0.0249
2017	14,676	12,339	358,266	0.0343
2018	20,900	17,100	?	?

14.0 Mineral Resource Estimate

Ms. Lane and Dr. Samari updated the April 15, 2009 mineral resource for the Chandalar Project (Barker, et al., 2009) to incorporate drilling completed in 2013 and 2017, as well as mining undertaken from 2009 through 2018.

The mineral resource estimate was completed under the direction of Hamid Samari, Senior Geologist, and Terre Lane, Principal of GRE and Qualified Persons. Resource modeling and estimation was carried out using Seequent Leapfrog® software.

14.1 Definitions

The mineral resource estimate presented herein conforms with US Guide 7 and the definitions adopted by the U.S. Securities and Exchange Commission (SEC) Item 102 of Regulation on S-K February 25, 2019, and Committee for Mineral Reserves International Reporting Standards (Sept. 23, 2016) ("CRIRSCO"), whereas:

> Mineral resource is a concentration or occurrence of 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 economic extraction. A mineral resource is a reasonable estimate of mineralization, taking into account relevant factors such as cut-off grade, likely mining dimensions, location or continuity, that, with the assumed and justifiable technical and economic conditions, is likely to, in whole or in part, become economically extractable. It is not merely an inventory of all mineralization drilled or sampled.
>
> Measured mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of conclusive geological evidence and sampling. The level of geological certainty associated with a measured mineral resource is sufficient to allow a qualified person to apply modifying factors, as defined in this section, in sufficient detail to support detailed mine planning and final evaluation of the economic viability of the deposit. Because a measured mineral resource has a higher level of confidence than the level of confidence of either an indicated mineral resource or an inferred mineral resource, a measured mineral resource may be converted to a proven mineral reserve or to a probable mineral reserve.
>
> Indicated mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of adequate geological evidence and sampling. The level of geological certainty associated with an indicated mineral resource is sufficient to allow a qualified person to apply modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Because an indicated mineral resource has a lower level of confidence than the level of confidence of a measured mineral resource, an indicated mineral resource may only be converted to a probable mineral reserve.
>
> 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. The

level of geological uncertainty associated with an inferred mineral resource is too high to apply relevant technical and economic factors likely to influence the prospects of economic extraction in a manner useful for evaluation of economic viability. Because an inferred mineral resource has the lowest level of geological confidence of all mineral resources, which prevents the application of the modifying factors in a manner useful for evaluation of economic viability, an inferred mineral resource may not be considered when assessing the economic viability of a mining project, and may not be converted to a mineral reserve.

14.2 Resource Estimation Procedures

The gold resource estimate is based on the current drill hole database and estimated mined-out volumes. Three-dimensional (3D) modeling of the mined-out volumes was completed with Leapfrog Geo, and estimation of mineral resources was completed using Leapfrog EDGE (Version 5.1).

Geostatistical analysis, block model estimation, and block model validation were completed using Leapfrog EDGE.

14.3 Data Used for the Estimation

14.3.1 Drill Holes

The mineral resource estimate incorporates assay results from 2007, 2013, and 2017 drilling on the project. Drill data provided by Goldrich and verified by Hamid Samari, included collar coordinates, lithology, and assay data. All holes were drilled vertically. This study uses 395 drill holes totaling 35,930 feet (104 2007 drill holes totaling 15,373.4 feet, 61 2013 drill holes totaling 6,253.7 feet, and 230 2017 drill holes totaling 14,303 feet), with an average depth of 90 feet per hole. The collar locations are projected in the local project coordinate system with planar and elevation units in feet. All downhole intervals are in feet.

Ms. Lane believes the drill hole assay data are sufficiently reliable to support the estimation of gold mineral resources.

14.3.2 Assay Data

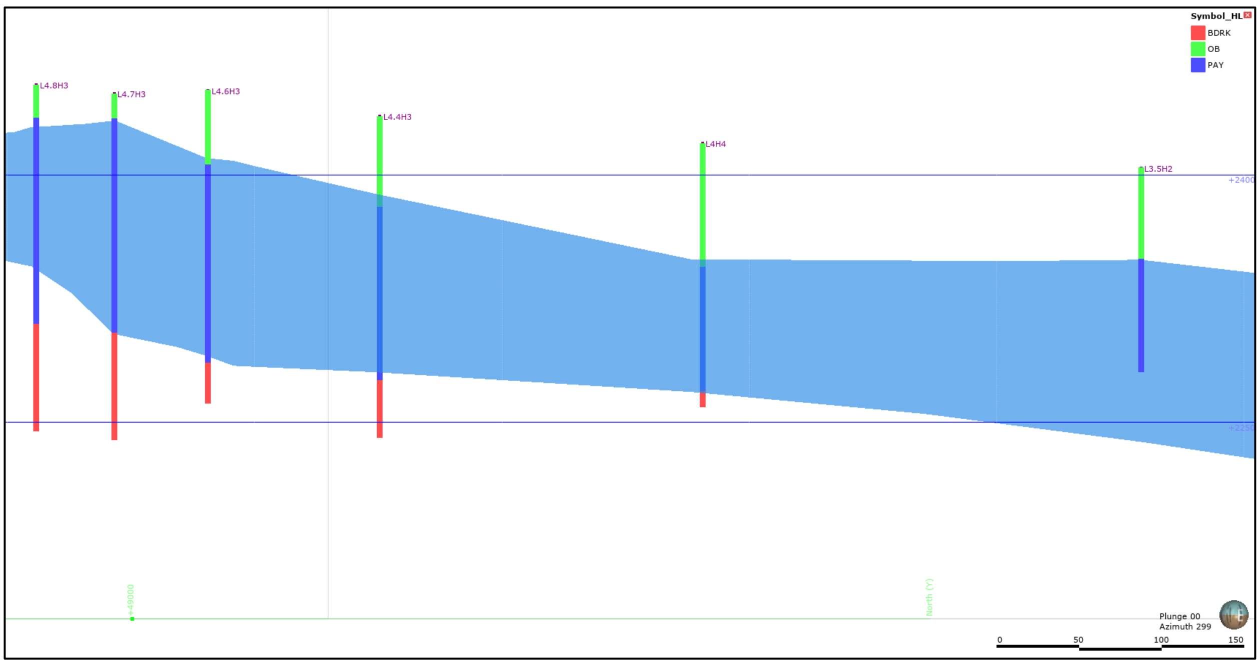
The Goldrich-provided database contained several grade fields, including one named “Grade_HL” and another named “composite au 2013,2017.” The assay data included hole ID and gold grade in oz/bcy. Samples from the 2007 and 2013 drill campaigns were conducted on 5-foot intervals. Samples from the 2017 drill campaign were conducted on 2-foot intervals. The 2007 and 2013 drilling samples (5,042 samples) were analyzed by Goldrich (contained in the Grade_HL field), and the 2017 drilling samples (7,671 samples contained in the composite au 2013,2017 field) were analyzed by NyacAU, of which 5,632 were used, resulting in 10,674 assays values in the complete data set.

Gold grades from the 2007 campaign were provided as 999 fine gold values. Grades for the 2013 and 2017 drilling were provided in raw gold with an 870 fineness. Ms. Lane converted the 2007 values to raw gold (870 fineness) to create a consistent data set of raw gold for the resource estimation. Drill intervals with missing assay values were set to 0 under the assumption that they were deemed unmineralized intervals by the geologist during the sample selection. Ms. Lane used raw gold in the resource modeling and used 840 fineness in the economic model (Section 21.0).

14.3.3 Lithology

The drill database contains a lithology field describing the formations as overburden, pay gravel, and bedrock for 143 of the 395 drill holes. This information was not used in the geologic model. However, Ms. Lane visually verified that the available lithology information compared well with the domain of the pay gravel provided by Goldrich. Figure 14-1 shows a typical cross-section where the lithology categories indicate that the pay gravel wireframe does not include overburden or bedrock.

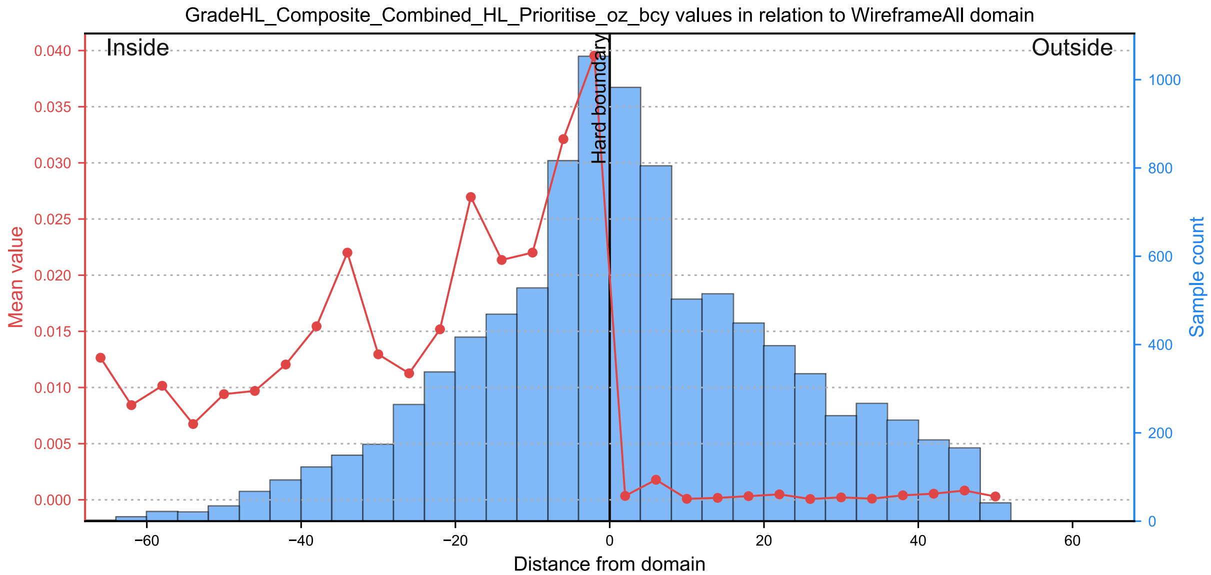
Figure 14-1: Comparison of Available Lithology with Estimation Domain



14.4 Domain Analysis

A Goldrich consulting geologist created a pay gravel wire frame solid using a 0.002 oz Au/bcy cutoff to describe the mineralized volume of the pay gravel. Ms. Lane reviewed the wire frame solid and used it to constrain the resource estimate and to prevent excessive dilution from material outside the pay gravel. Figure 14-2 shows the boundary analysis for the pay gravel, clearly showing a hard boundary outside of the wireframe with elevated gold values contained within.

Figure 14-2: Boundary Analysis, Pay Gravel Wireframe



14.5 Existing Topography and Mined-Out Volumes

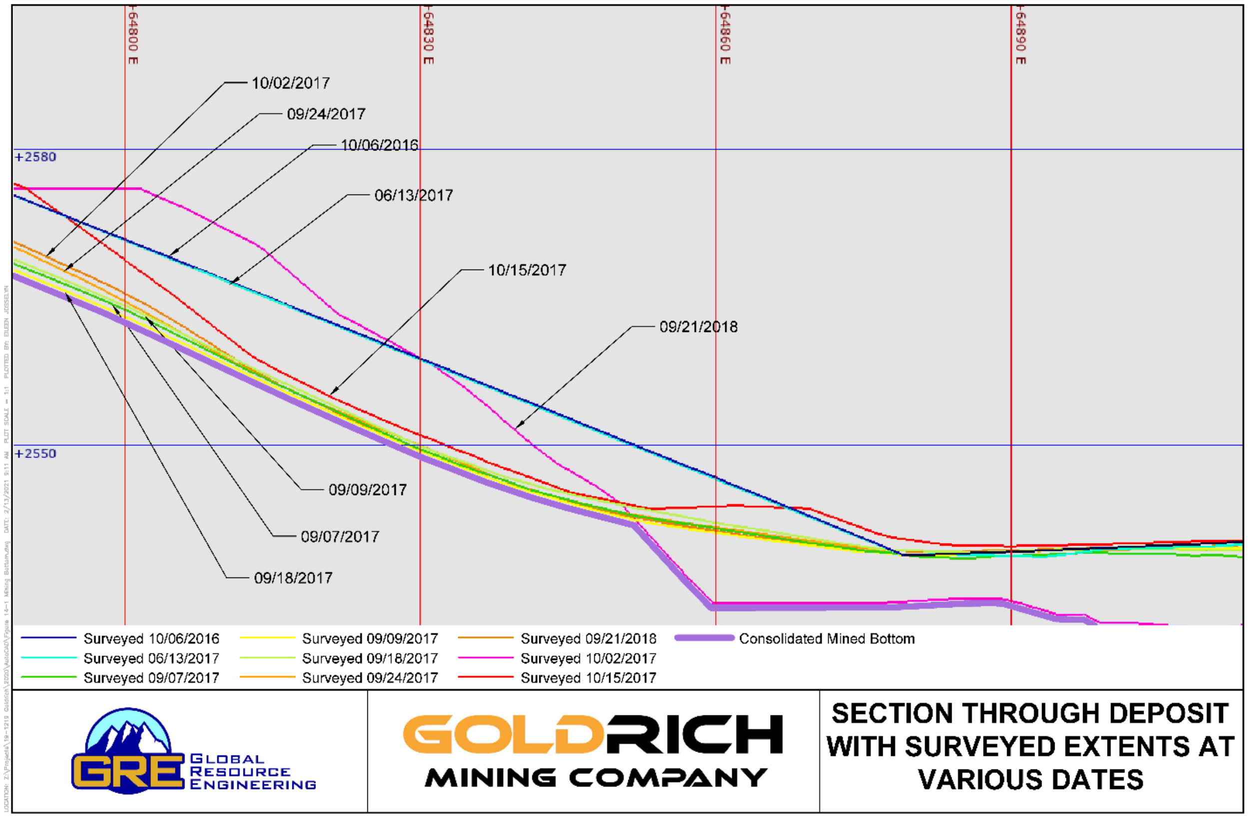
Goldrich provided Ms. Lane with a detailed FODAR survey of the Little Squaw project area completed in 2019, subsequent to all previous GNP mining activity, which ceased after the 2018 season. The FODAR survey was completed in UTM WGS84 coordinates. Surveys conducted by GNP during operations were completed in a local coordinate system. Goldrich was unable to provide Ms. Lane with a conversion from UTM coordinates to the locate coordinate system used by GNP. GRE does not have licensed surveyors on staff but attempted to use available software to translate the FODAR survey to the local coordinate system, which involved converting it from meters to feet and translating x, y, and z coordinates to the local system. The translation is approximate. The final topographic surface Ms. Lane used stitched together the final 2018 survey completed by GNP on 21 September 2018, which is limited to the areas immediately surrounding the mined areas, and the translated LIDAR survey. This stitched together surface constitutes a small risk to the estimated resources, but because the 2018 surface would be lower than the 2019 surface due to backfilling that may have occurred in 2019 and due to the fact that there was no mining conducted in 2019, Ms. Lane believes the estimated remaining in-situ resource would not change with the 2019 topographic surface but that there could be additional waste backfilled above the estimated resource.

Ms. Lane was notified that GNP had previously backfilled some mined-out areas; therefore, neither the 2018 nor 2019 topographic survey would accurately establish mined-out areas of the project. Goldrich provided GRE with a series of AutoCAD dwg files showing contours within the mined areas representing as-built topography as of the following dates:

- 9 September 2015
- 6 October 2016
- 8 October 2016
- 10 October 2016
- 13 June 2017
- 26 June 2017
- 19 August 2017
- 9 September 2017
- 18 September 2017
- 24 September 2017
- 2 October 2017
- 15 October 2017
- 21 September 2018

Goldrich indicated to GRE that these contour files represented the extents of mining as of those dates. Ms. Lane created a surface from each of the contour files, then created a consolidated minimum surface to approximate the total mined extents of the Project. Figure 14-3 illustrates the method used to define the previous bottom of mining extents.

Figure 14-3: Method for Estimating Bottom of Mined-out Volumes at Chandalar Mine Project



14.6 Assay Compositing and Outliers

The assay data was composited to a length of 10 feet. Any residual end composite samples less than five feet in length were added to the previous composite sample interval. The maximum assay interval was 10 feet, which eliminates the risk of dividing single assays into multiple composites. Figure 14-4 shows histograms of the uncomposited and composited sample interval lengths.

The statistics of both the composited and uncomposited samples within the pay gravel wire frame are shown in Table 14-1.

Ms. Lane analyzed the composite values and determined that capping was not necessary. Figure 14-5 shows a log probability plot of the composited values showing a single linear trend.

Figure 14-4: Sample Interval Lengths, Uncomposited and Composited

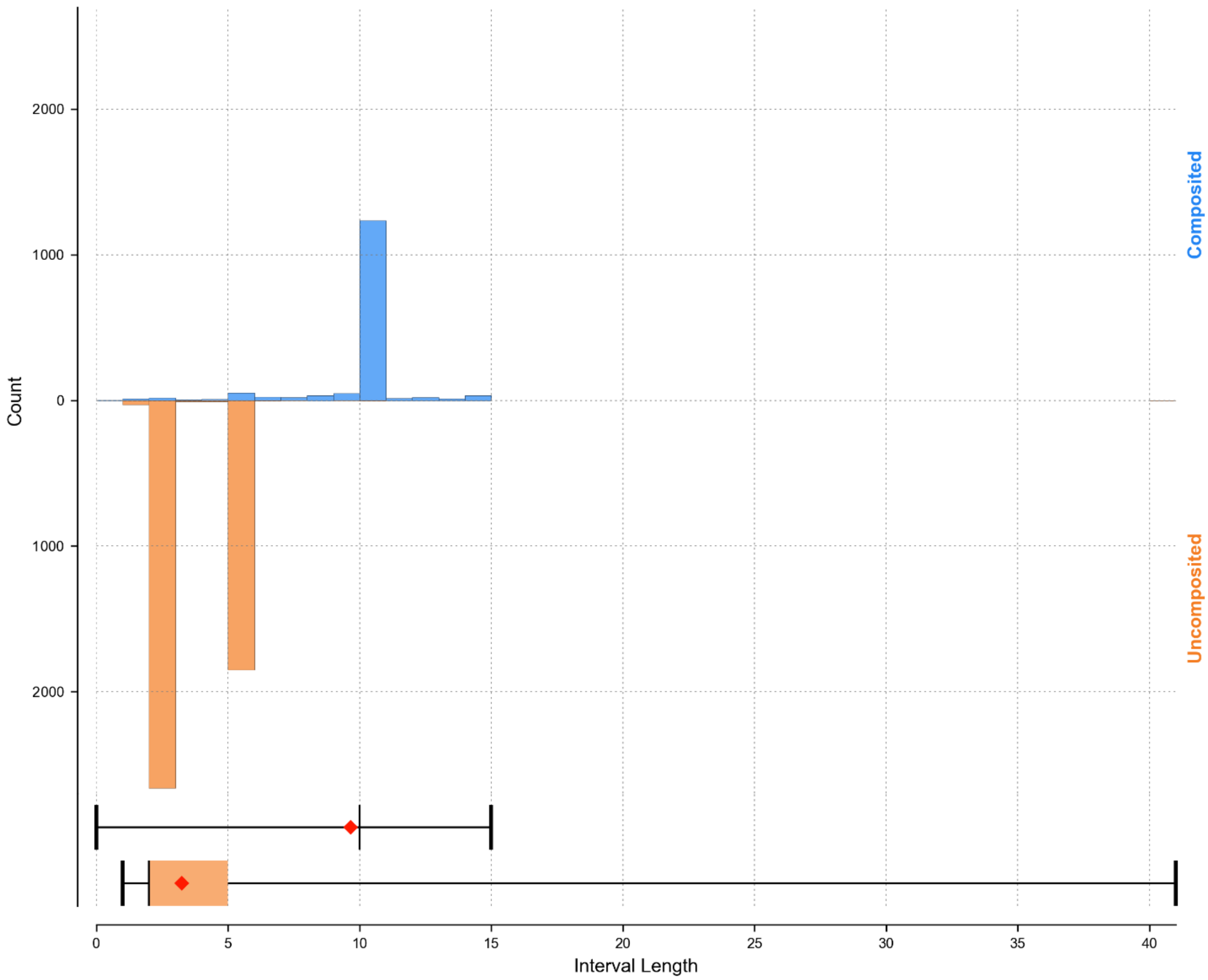
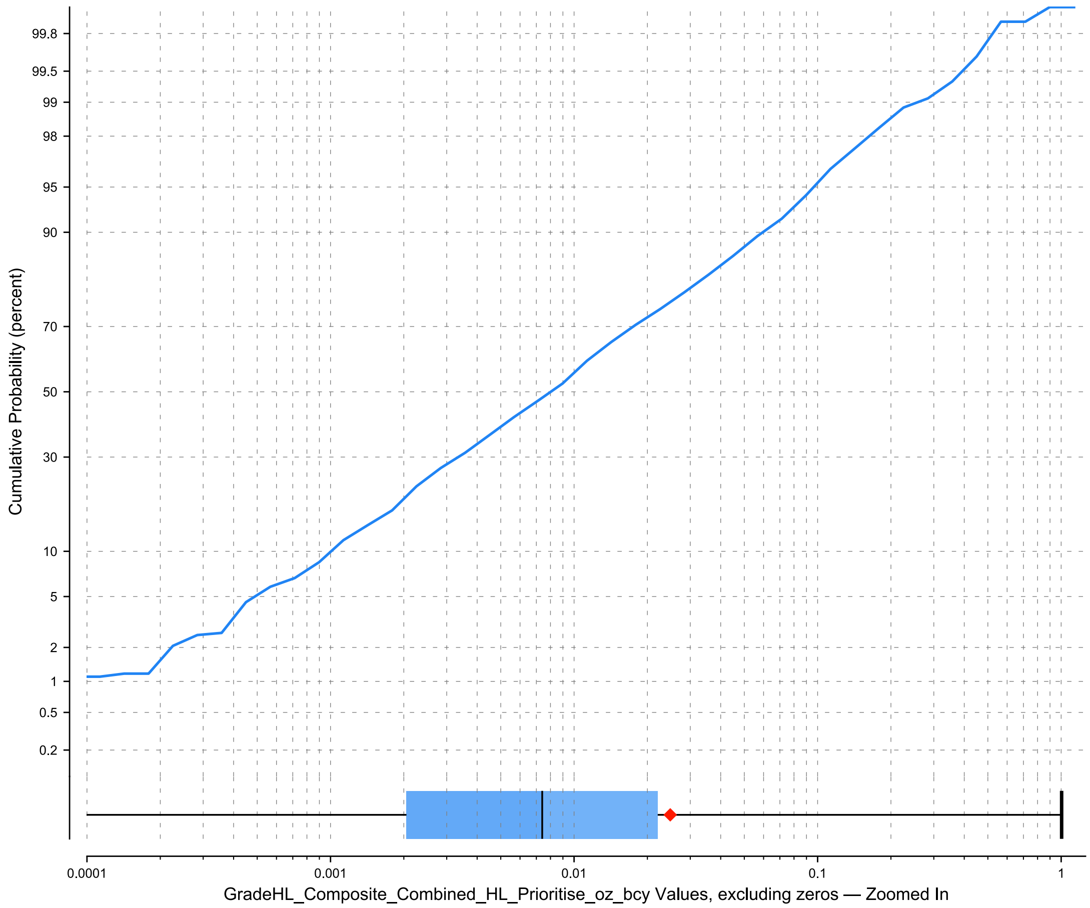


Table 14-1: Sample Statistics for Uncomposited and Composited Intervals

	Composited (Raw Gold/bcy)	**Uncomposited** (Raw Gold/bcy)
Count	1,534	4,575
Length	14,826	14,835
Mean	0.025	0.025
SD	0.060	0.093
CV	2.42	3.76
Variance	0.0036	0.0087
Minimum	0.0	0.0
Q1	0.002	0.0
Q2	0.007	0.004
Q3	0.022	0.016
Maximum	1.005	2.310

Figure 14-5: Cumulative Probability Plot of Composited Sample Values at Chandalar Mine



14.7 Density

The assay values for this deposit are provided in raw gold ounces per bank cubic yard because volume results rather than tonnage results are appropriate for placer mining operations. Therefore, the mineralized material density is not applicable to the resource estimation.

14.8 Block Model Parameters

The block model parameters for Chandalar Mine are presented in Table 14-2.

Table 14-2: Chandalar Mine Block Model Parameters

Direction	Block Size (feet)	Start	End	Number
Easting	20	62650	69810	208
Northing	20	40750	48750	400
Elevation (AMSL)	5	3150	2050	220

A block height of 5 feet was defined based on mining equipment expected to be used in an open pit scenario.

14.9 Variography

As is typical with placer gold deposits, the coarse nature of the gold causes a very high nugget effect in variography, overriding any structural characteristics of the variogram. Ms. Lane used inverse distance to the power of 3 to estimate grade.

14.10 Estimation Methodology

Block grades were estimated by a single pass estimator using the inverse distance cubed (ID3) method and the ellipsoid search parameters shown in Table 14-3. The ellipsoid is oriented along the strike of the main valley containing the deposit.

Table 14-3: Chandalar Mine Estimation Ellipsoid Parameters

Parameter	Value
Maximum Ellipsoid Range	300
Intermediate Ellipsoid Range	150
Minimum Ellipsoid Range	30
Dip	7
Dip Azimuth	18
Pitch	90
Minimum Samples	1
Maximum Samples	8
Boundary Condition	Hard

14.11 Block Model Validation

Validation of the estimated block grades was completed by:

- Visual comparisons of composite sample grades to estimated block values across the deposit
- Statistical comparison of the composites, nearest neighbor block estimate, and ID3 block estimate
- Swath plots comparing average composite sample values with average estimated block grades along east, north, and elevation orientations

14.12 Production reconciliation using the mined-out surfaces from 2016 through 2018

14.12.1 Visual Comparison

Estimated raw gold block grades were visually compared to the composite values for various detailed sections across the deposit. This review confirmed that the supporting composite sample grades closely match the estimated block values. Figure 14-6 displays a representative long section, and Figure 14-7 shows a representative cross section, where the color scale represents Au raw opy.

Figure 14-6: Chandalar Mine Block Model and Composite Sample Gold Values, Long Section

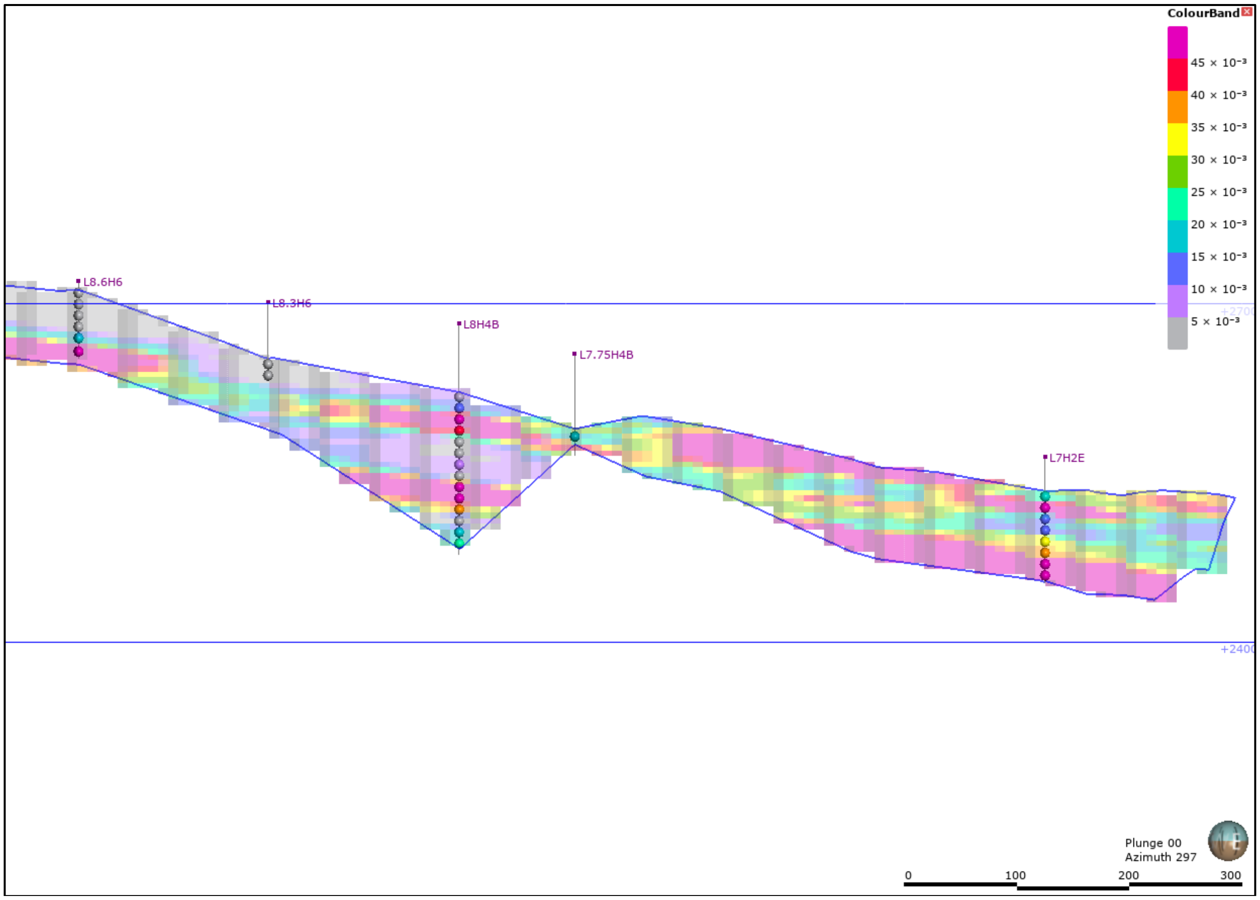
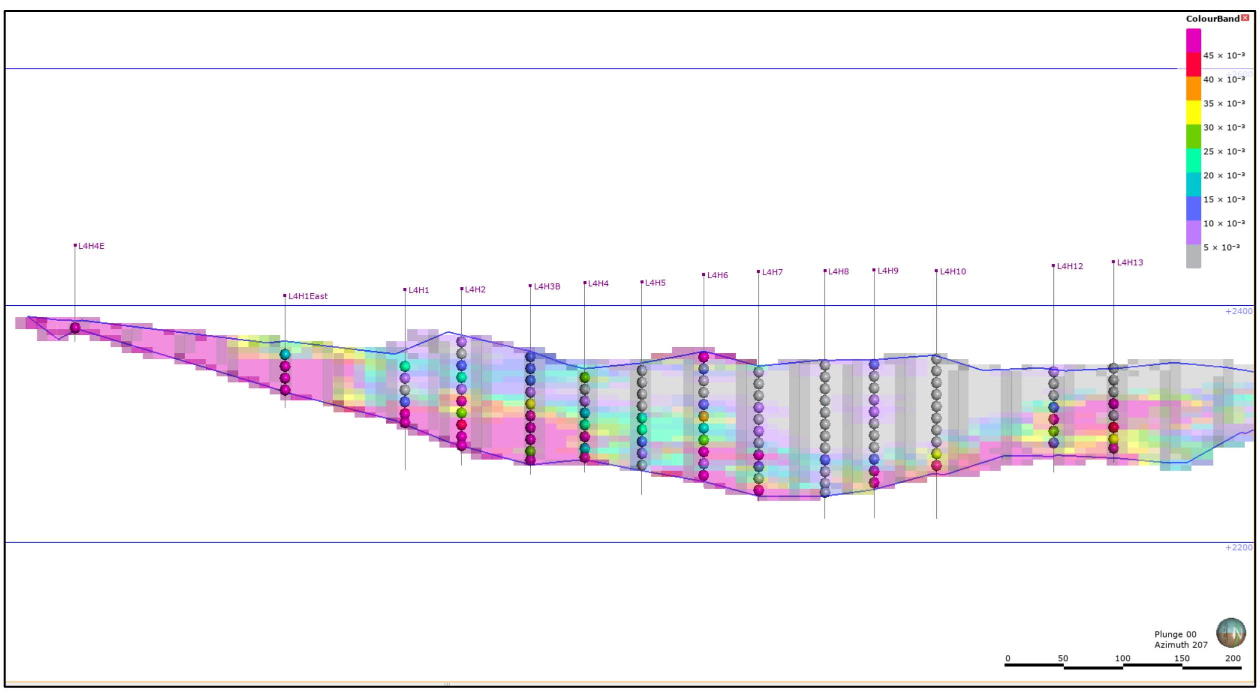


Figure 14-7: Chandalar Mine Block Model and Composite Sample Gold Values, Cross Section



14.12.2 Statistical Comparison

Table 14-4 shows the statistical comparison between the composites, nearest neighbor (NN) block estimate, and ID3 block estimate. The NN mean provides an estimate of the declustered composite mean, which limits the influence of close-proximity samples and better represents the overall mean of the mineralized body. The ID3 estimate has a similar mean, quartile values, and maximum, indicating the ID3 estimate corresponds well to the overall deposit block distribution. The coefficient of variation for the ID3 estimate is reduced from the NN estimate, indicating a degree of smoothing in the block model.

Table 14-4: Statistical Comparison, Composites, NN, & ID3 (raw oz/bcy)

Composite Parameter	Composites	Block Model Parameters	NN	ID3
	Weighted Value		Weighted Value	
Count	1,534	Block Count	132,362	132,362
Length	14,826	Volume	264,724,000	264,724,000
Mean	0.025	Mean	0.023	0.023
SD	0.060	SD	0.054	0.035
CV	2.419	CV	2.376	1.507
Variance	0.004	Variance	0.003	0.001
Minimum	0.000	Minimum	0.000	0.000
Q1	0.002	Q1	0.003	0.006
Q2	0.007	Q2	0.008	0.012
Q3	0.022	Q3	0.020	0.027
Maximum	1.005	Maximum	1.005	0.998

14.12.3 Swath Plots

Swath plots provide a graphical method of comparing composite grades with the NN and ID3 block model estimates. Figure 14-8 through Figure 14-10 present swath plots along the X-axis, Y-axis, and Z-axis of the block model. The volume of blocks is shown as a bar graph in the background of each figure. Like the statistical comparison, the swath plots show good correlation of grade values between NN and ID3 block estimates. The plot along the X-axis is well aligned with the search orientation and therefore shows the best correlation between the composites and the block grades.

Figure 14-8: Swath Plot along X-axis (raw oz/bcy)

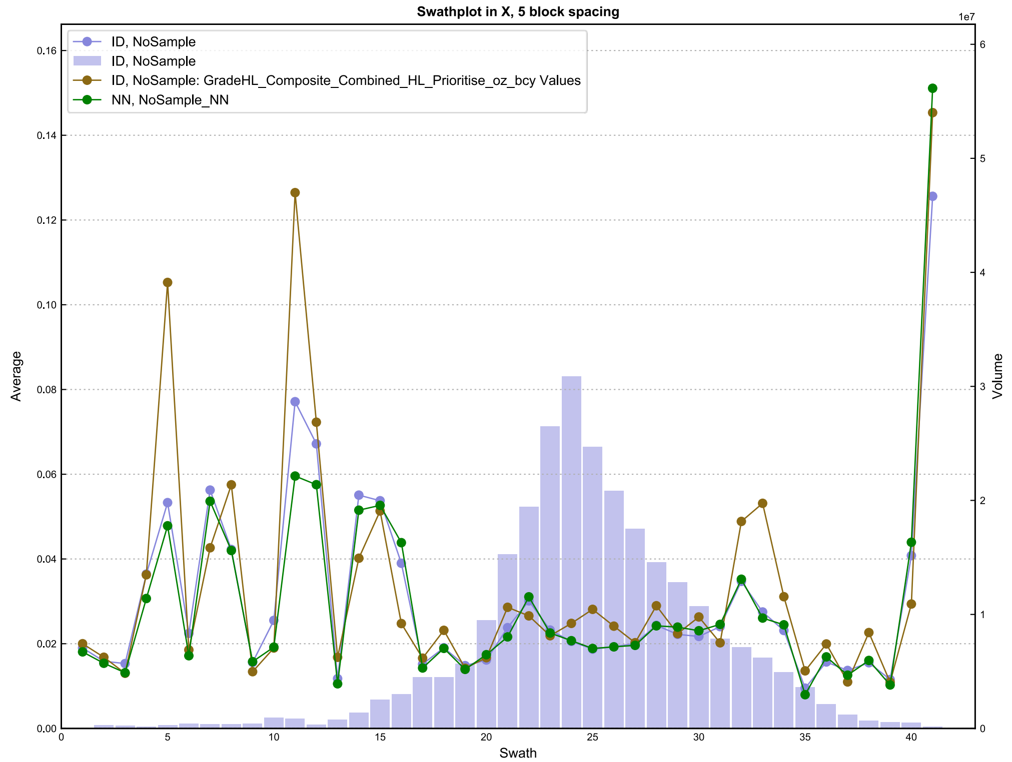


Figure 14-9: Swath Plot along Y-axis (raw oz/bcy)

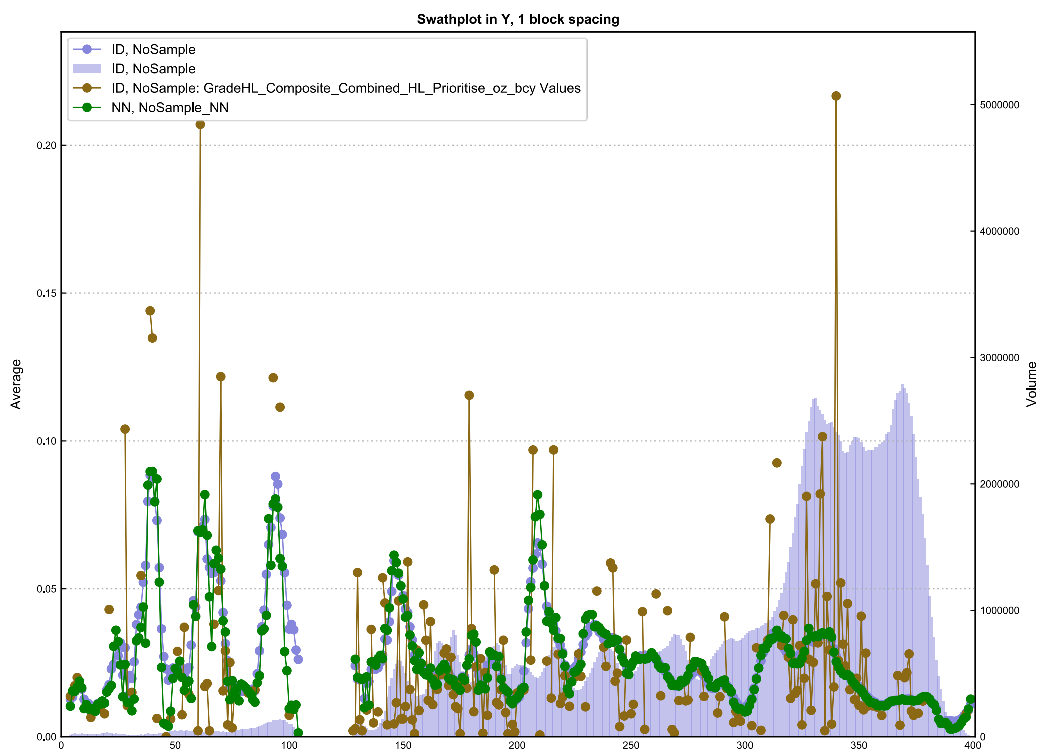
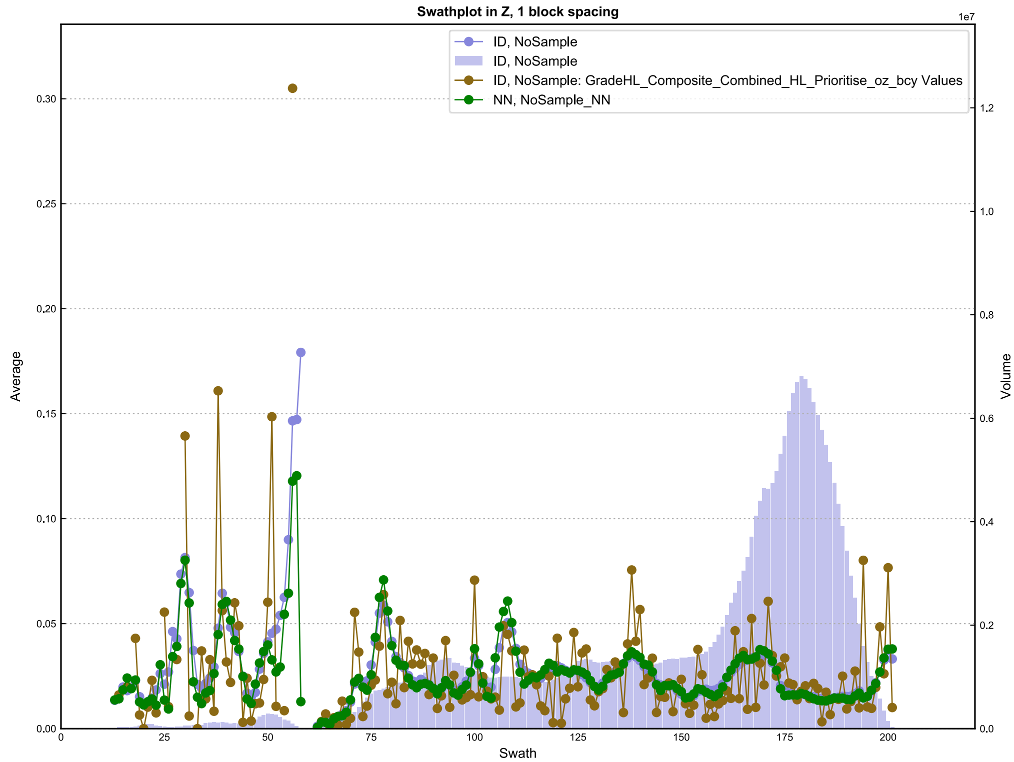


Figure 14-10: Swath Plot along Z-axis (raw oz/bcy)



14.12.4 Production Reconciliation

Ms. Lane completed a production reconciliation to the ID3 block model to determine the accuracy of the estimation. The reconciliation compared the reported mined raw ounces from 2015 through 2018 with the estimated raw ounces using the consolidated mined-out surface for the same time period. The block model estimates 47,700 raw ounces (at an assumed cutoff grade of 0.002 opy) compared to a reported production of 53,900 raw ounces, or actual production was 13% greater than the block model estimate (Table 14-5).

Table 14-5: Reconciliation of Mine Block Model to Actual Gold Production

Year	Block Model Estimate (at assumed cutoff grade of 0.002 opy)	Approximate Actual Production	Actual Oz of Raw Gold Greater/(Less) Than Estimate	% Actual Oz of Raw Gold Greater/(Less) Than Estimate
2009 to 2015	7,300	8,150	500	12%
2016	13,700	10,200	-3,500	-26%
2017	10,700	14,680	4,300	37%
2018	16,000	20,360	4,900	27%
Total	**47,700**	**53,390**	**6,200**	12%

14.13 Mineral Resource Classification

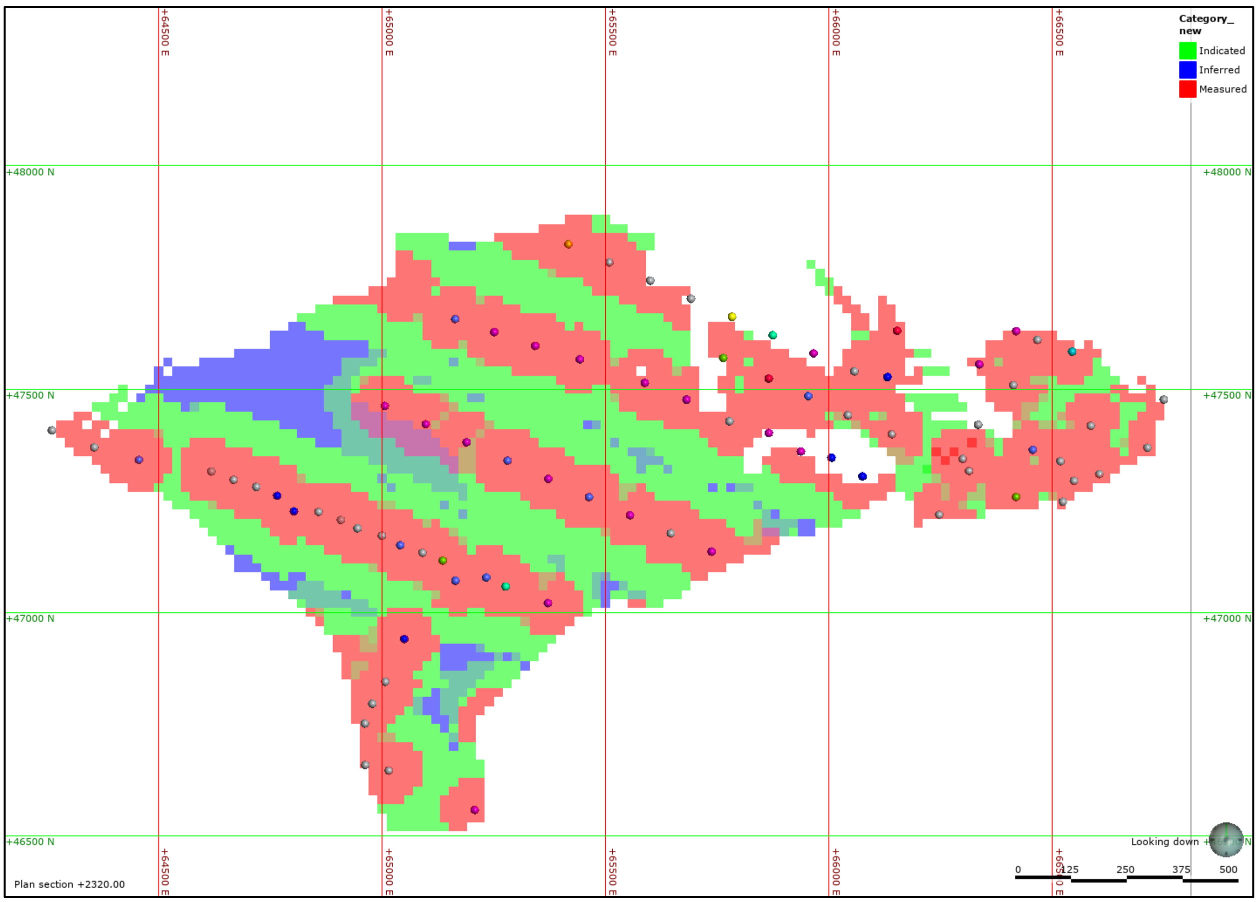
Block model quantities and grade estimates for the Chandalar Mine placer deposit were classified according to the 17 CFR §229.1300 – (Item 1300) Definitions. Mineral resources were estimated in conformity with generally accepted 17 CFR §229.1300 – (Item 1300) Definitions.

Remaining mineral resources are classified as Measured, Indicated, or Inferred based on the minimum distance to a composite value as follows:

- Measured: 0-75 feet
- Indicated: 76-150 feet
- Inferred: 151-400 feet

Figure 14-11 shows a plan view of the resource classification at the 2320-foot level of the block model.

Figure 14-11: Measured, Indicated, and Inferred Resources with Composite Samples



14.14 Constrained Mineral Resource

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic, and other factors. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Mineral reserves can only be estimated based on the results of an economic evaluation as part of a Preliminary Feasibility Study or Feasibility Study. As a result, no mineral reserves have been estimated as part of this study. There is no certainty that all or any part of the mineral resources will be converted into a mineral reserve.

The requirement, “reasonable prospects for eventual economic extraction,” generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at a cutoff grade considering appropriate extraction scenarios and processing recoveries. To meet this requirement, Ms. Lane considered that major portions of the Chandalar Mine deposit are amenable for open pit extraction.

To determine the quantities of material offering “reasonable prospects for eventual economic extraction” by an open pit, Ms. Lane constructed open pit scenarios developed from the resource block model estimate using Whittle Lerchs-Grossman miner software. Reasonable mining assumptions were applied to evaluate the portions of the block model (Measured, Indicated, and Inferred blocks) that could be “reasonably expected” to be mined from an open pit. The optimization parameters presented in Table 14-6 were selected based on experience and benchmarking against similar projects. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cutoff grade. Ms. Lane considers that the blocks located within the resulting conceptual pit envelope show “reasonable prospects for economic extraction” and can be reported as a mineral resource.

Table 14-6: Chandalar Mine Resource Parameters for Conceptual Open Pit Optimization

Parameter	Unit	Values
Metal Price	US$/oz gold	$1,600.00
Gold Fineness	%	84.00%
Mining cost	US$/bcy	$4.50
Process cost and Administrative cost	US$/bcy	$7.25
Pit slope	degrees	45

The reader is cautioned that the results from the pit optimization are used solely for testing the “reasonable prospects for eventual economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves. There are presently no mineral reserves on the project.

The Chandalar Mine pit-constrained Mineral Resource Estimate at a 0.004 opy cutoff grade is shown in Table 14-7. The cutoff is the life of mine process cost $3.92, camp cost $0.85, and transportation cost $1.65 divided by the gold price $1,600, which equals 0.004 oz/BCY gold or 0.0048 raw gold. The operating cost during the middle of the operating life are lower; therefore, GRE selected 0.004 oz/BCY raw gold as a reasonable blend cutoff grade.

Table 14-7: Mineral Resource Statement for the Chandalar Mine

Classification	Resource Volume (1000s bcy)	Raw Gold Grade (t.oz./bcy)	Raw Gold (1000s t. oz)	Fine Gold (1000s t. oz)
Measured	2,609	0.0302	79	69
Indicated	2,188	0.0265	58	50

Measured & Indicated	4,797	0.0285	137	119
Inferred	771	0.0245	19	16

1) The effective date of the Mineral Resource is May 31, 2021.
2) The Qualified Persons for the estimate are Hamid Samari and Terre Lane of GRE.
3) Mineral Resources are inclusive of Mineral Reserves; Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
4) Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.
5) The Mineral Resource is constrained by a 0.002 raw troy ounce per bank cubic yard grade shell and a 0.004 raw troy ounce per bank cubic yard cutoff (840 fineness) at an assumed gold price of 1,600 $/tr oz, assumed mining cost of 4.50 $/bcy, assumed processing and administrative cost of 7.25 $/bcy, an assumed gold purity of 84%, and pit slopes of 45 degrees. These costs are preliminary estimates (prior to economic analysis).

14.15 Grade Sensitivity to Gold Cutoff

The mineral resources reported for the Chandalar Mine deposit are sensitive to the selection of the reporting gold cutoff grade. To illustrate this sensitivity, the block model gold quantities and grade estimates are presented at different cutoff grades within the conceptual pit used to constrain the mineral resources (Table 14-8). The reader is cautioned that the information presented in the table should not be misconstrued as a Mineral Resource Statement.

The mineral resources are constrained by a 0.002 raw troy ounce/bank cubic yard grade shell and a 0.004 raw troy ounce/bank cubic yard cutoff. The cutoff grades shown in Table 14-8 are marginal cutoff grades, i.e., they include processing and G&A costs but not mining costs.

Table 14-8: Chandalar Mine Deposit Mineral Resource Sensitivity

Classification	Cutoff	Resource Volume (1000s bcy)	Raw Gold Grade (t.oz./bcy)	Raw Gold (1000s t. oz)	Fine Gold (1000s t. oz)
Measured	0.002	2,819	0.0282	79	69
	0.004	**2,609**	**0.0302**	**79**	**69**
	0.006	2,385	0.0326	78	68
	0.008	2,149	0.0354	76	66
	0.01	1,949	0.0381	74	65
	0.012	1,200	0.0528	63	55
Indicated	0.002	2,309	0.0253	58	51
	0.004	**2,188**	**0.0265**	**58**	**50**
	0.006	2,048	0.0280	57	50
	0.008	1,871	0.0300	56	49
	0.01	1,681	0.0323	54	47
	0.012	1,006	0.0444	45	39
Measured & Indicated	0.002	5,128	0.0269	138	120
	0.004	**4,797**	**0.0285**	**137**	**119**
	0.006	4,433	0.0305	135	117
	0.008	4,020	0.0329	132	115
	0.01	3,630	0.0354	129	112
	0.012	2,206	0.0490	108	94
Inferred	0.002	832	0.0230	19	17

Classification	Cutoff	Resource Volume (1000s bcy)	Raw Gold Grade (t.oz./bcy)	Raw Gold (1000s t. oz)	Fine Gold (1000s t. oz)
	0.004	**771**	**0.0245**	**19**	**16**
	0.006	712	0.0262	19	16
	0.008	641	0.0283	18	16
	0.01	583	0.0302	18	15
	0.012	367	0.0395	15	13

15.0 Mineral Reserve Estimates

There are no mineral reserve estimates for the Chandalar Mine project.

16.0 Mining Methods

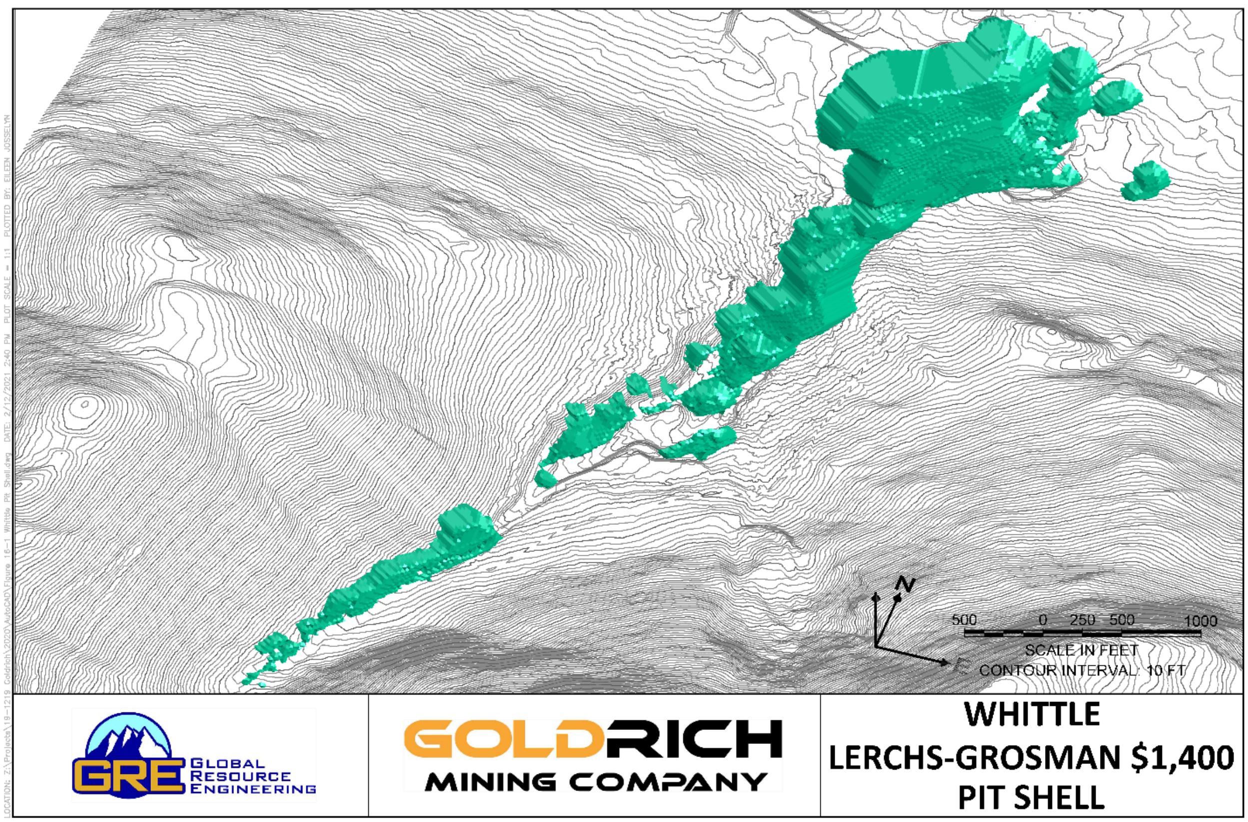
Conventional open pit mining methods using drill, blast, load, and haul mining are applicable to the Chandalar Mine because the gravel is frozen (permafrost). Previous operators successfully used these methods.

Mine scheduling and optimization were conducted as described in the following subsections.

16.1 Designed Pit

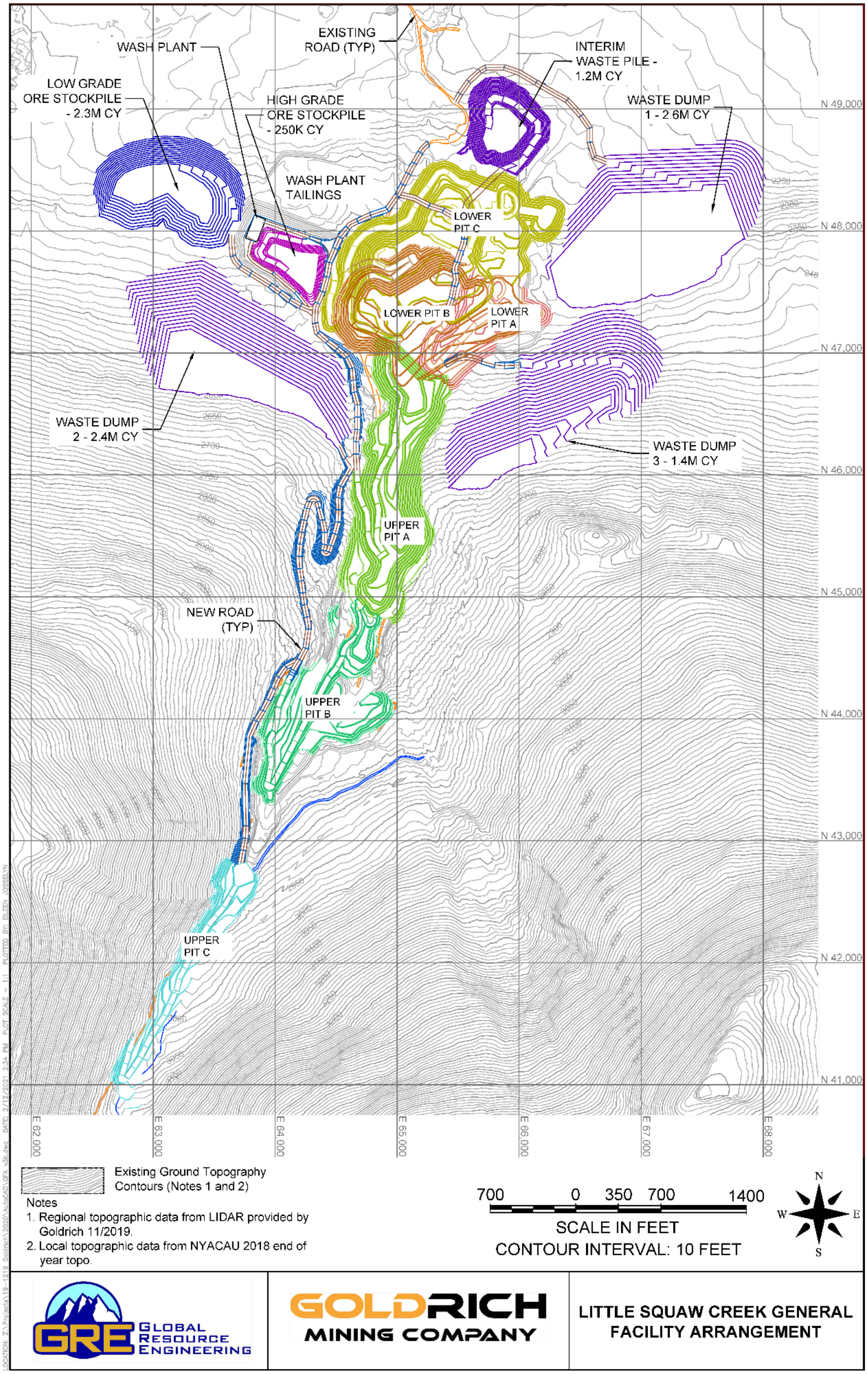
As stated in Section 14.14, Ms. Lane generated pit shells using Whittle Lerchs-Grossman software using reasonable mining assumptions as identified in Table 14-6. Twenty-eight pit shells were generated using gold prices ranging from $300/Au raw ounce to $3,000/Au raw ounce, in increments of $100/Au raw ounce. After a preliminary review of the pit shell data economics, Ms. Lane focused on the $1,400/Au raw ounce pit shell, which showed the best overall preliminary economic results. Figure 16-1 shows the $1,400/Au raw ounce Whittle pit shell.

Figure 16-1: Chandalar Mine $1,400/oz Raw Gold Whittle Pit Shell



The pit shell was imported into Geovia GEMS software, where it was segregated into six separate pits or pit pushbacks designated as Lower Pit A, Lower Pit B, Lower Pit C, Upper Pit A, Upper Pit B, and Upper Pit C. Each pit was designed with haul roads. Figure 16-2 shows the six ultimate pits/pushbacks.

Figure 16-2: Chandalar Mine Designed Pit



16.2 Reported Resources

Resources for each designed pit/pushback were reported out of GEMS by bench and with codes identifying mined out status (i.e., blocks that had been previously mined out and backfilled were identified as waste), average grade of processable material, and resource category (i.e., Measured, Indicated, and Inferred). All resource categories were treated equally for the purposes of this IA.

16.3 Options Evaluated

Ms. Lane imported the GEMS resource output into a Multi-Scenario Economic Model. Thirty scenarios were evaluated, with the following parameters:

- Phasing Sequence:

Phasing Sequence Designation	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Phase 6
1	Lower Pit A	Upper Pit A	Upper Pit B	Lower Pit B	Lower Pit C	Upper Pit C
2	Upper Pit A	Upper Pit B	Upper Pit C	Lower Pit A	Lower Pit B	Lower Pit C
3	Upper Pit A	Lower Pit A	Lower Pit B	Lower Pit C	Upper Pit B	Upper Pit C

- Cutoff Grade Options: The pits were evaluated at five mining cutover grades: 0.004 opy, 0.006 opy, 0.008 opy, 0.010 opy, and 0.012 opy. Material between the economic cutoff grade of 0.004 opy and the designated mining cutoff grade would be considered low-grade material and stockpiled for processing after processing of all high-grade material.
- Production Rate Options:
  - 4,000 bcy per day for the first year (using one 4,000 bcy/day wash plant) and 8,000 bcy per day (using two 4,000 bcy/day wash plants) for all subsequent years (the “ramp-up production” option)
  - 4,000 bcy per day for all years.

16.4 Evaluation

The economic model for the mine evaluated the 30 cases to optimize the mine planning and design. Based on the economic analysis of all 30 cases, Ms. Lane selected sequence 2 at a cutover grade of 0.012 opy, and the ramp-up production option. All further references to “the ultimate pit” or the “base case” in this report are referring to sequence 2 at a cutover grade of 0.012 opy, and the ramp-up production option.

The resources contained within the base case designed pit are shown in Table 16-1.

Table 16-1: Chandalar Mine Base Case Designed Pit Resource

Phase	High-Grade Material (million bcy)	Low-Grade Material (million bcy)	Waste (million bcy)	Total (million bcy)	High Grade Au Raw (1000 oz)	High Grade Au Raw Grade (opy)	Low Grade Au Raw (1000 oz)	Low Grade Au Raw Grade (opy)	Stripping Ratio
Phase 1	0.99	0.37	1.48	2.84	37.85	0.038	2.91	0.008	1.08
Phase 2	0.17	0.06	0.48	0.70	7.19	0.043	0.44	0.008	2.14
Phase 3	0.08	0.02	0.17	0.27	4.15	0.053	0.16	0.009	1.82

Phase	High-Grade Material (million bcy)	Low-Grade Material (million bcy)	Waste (million bcy)	Total (million bcy)	High Grade Au Raw (1000 oz)	High Grade Au Raw Grade (opy)	Low Grade Au Raw (1000 oz)	Low Grade Au Raw Grade (opy)	Stripping Ratio
Phase 4	0.20	0.02	0.44	0.65	15.95	0.081	0.13	0.008	2.06
Phase 5	1.02	0.46	2.55	4.03	34.68	0.034	3.55	0.008	1.72
Phase 6	1.14	0.96	2.76	4.86	31.55	0.028	7.70	0.008	1.31
Total	**3.59**	**1.89**	**7.88**	**13.36**	**131.37**	**0.037**	**14.89**	**0.008**	**1.44**

16.5 Mine Schedule

A preliminary mining schedule was generated from the base case pit resource estimate. Ms. Lane used the following assumptions to generate the schedule:

- Process Material Production Rate: 4,000 bcy per day (ypd) for the first year (with one wash plant) and 8,000 bcy per day for subsequent years (with two wash plants)
- Mine Operating Days per Week: 7
- Mine Operating Period per Year: All of first three quarters (January 1 through October 31), with processing of mined material occurring only between May 17 and September 13 (120 days).
- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 12

Additional design work and phasing of the pits/pushbacks will likely better balance the annual process and waste stripping and metal production. Likewise, adding more blasting and waste stripping days to the seasonal schedule will help smooth out the production schedule.

In addition to the change in production rate from 4,000 ypd to 8,000 ypd after the first year, Ms. Lane included a gradual ramp-up to full production during the first year as shown in Table 16-2:

Table 16-2: Processing Ramp Up

Year/Month	Percent of Production
Year 1/first month	25%
Year 1/second month	50%
Year 1/third month	75%
Year 1/remaining months	100%

Ms. Lane also included a gradual ramp-up to full production of the second wash plant in year two over a period of 2 months.

High-grade material stockpiled during the first- and second-year ramp ups to full production and low-grade stockpiled material was scheduled to be processed after completion of regularly scheduled high-grade material processing.

Pre-stripping of waste was included if either of the following criteria were true: 1) waste occurred on a bench that had no corresponding process material or 2) the tonnage of waste on a bench exceeded 12 times the tonnage of process material on that bench. The mining rate for pre-strip benches was set to two times the processable material mining rate.

For all other benches, all waste on a bench was scheduled to be mined over the same duration as the process material on that bench. Ms. Lane then adjusted the waste removal schedule, keeping it within the same calendar year, but possibly in earlier quarters, to even out the mining equipment fleet.

The mining schedule is summarized by year in Table 16-3 and Figure 16-3.

Table 16-3: Chandalar Mine Base Case Mine Schedule Summary

Pit	Year 1	Year 2	Year 3	Year 4	Year 5	Total
High Grade Material bcy (millions)						
Phase 1	0.45	0.54				0.99
Phase 2		0.17				0.17
Phase 3		0.08				0.08
Phase 4		0.18	0.02			0.20
Phase 5		0.03	0.95	0.04		1.02
Phase 6			0.04	0.92	0.18	1.14
Total	**0.45**	**0.99**	**1.00**	**0.97**	**0.18**	**3.59**
Low Grade Material bcy (millions)						
Phase 1	0.13	0.24				0.37
Phase 2		0.06				0.06
Phase 3		0.02				0.02
Phase 4		0.01	0.01			0.02
Phase 5		0.03	0.43	0.00		0.46
Phase 6			0.02	0.90	0.05	0.96
Total	**0.13**	**0.36**	**0.46**	**0.90**	**0.05**	**1.89**
Waste bcy (millions)						
Phase 1	0.84	0.63				1.48
Phase 2		0.48				0.48
Phase 3		0.17				0.17
Phase 4		0.42	0.02			0.44
Phase 5		1.61	0.94	0.00		2.55
Phase 6			1.74	0.93	0.08	2.76
Total	**0.84**	**3.32**	**2.70**	**0.93**	**0.08**	**7.88**
High Grade Au Raw Ounces (1000s)						
Phase 1	16.65	21.20				37.85
Phase 2		7.19				7.19
Phase 3		4.15				4.15
Phase 4		14.55	1.40			15.95
Phase 5		0.74	32.38	1.56		34.68
Phase 6			0.78	25.78	4.98	31.55
Total	**16.65**	**47.83**	**34.57**	**27.35**	**4.98**	**131.37**
Low Grade Au Raw Ounces (1000s)						
Phase 1	1.03	1.87				2.91
Phase 2		0.44				0.44
Phase 3		0.16				0.16
Phase 4		0.09	0.04			0.13

Pit	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Phase 5		0.20	3.33	0.02		3.55
Phase 6			0.19	7.07	0.45	7.70
Total	**1.03**	**2.76**	**3.57**	**7.09**	**0.45**	**14.89**

Figure 16-3: Chandalar Mine Mine Schedule Summary

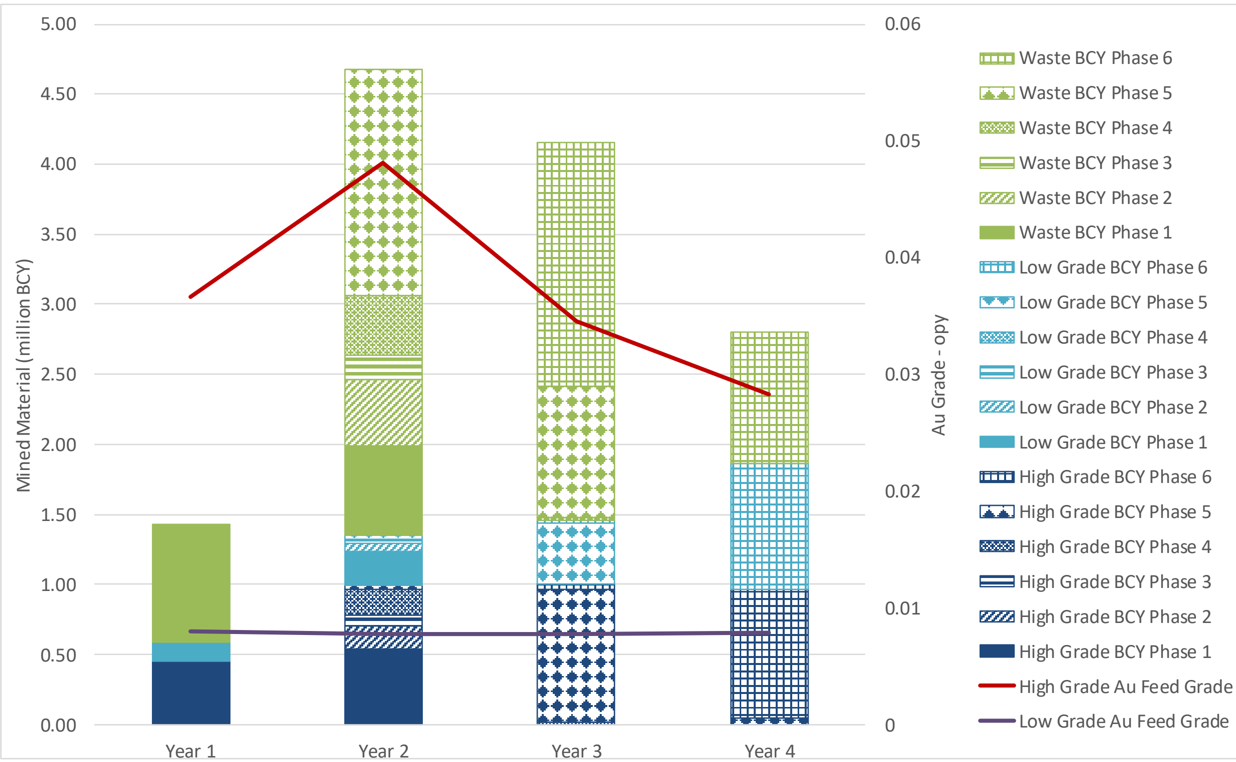


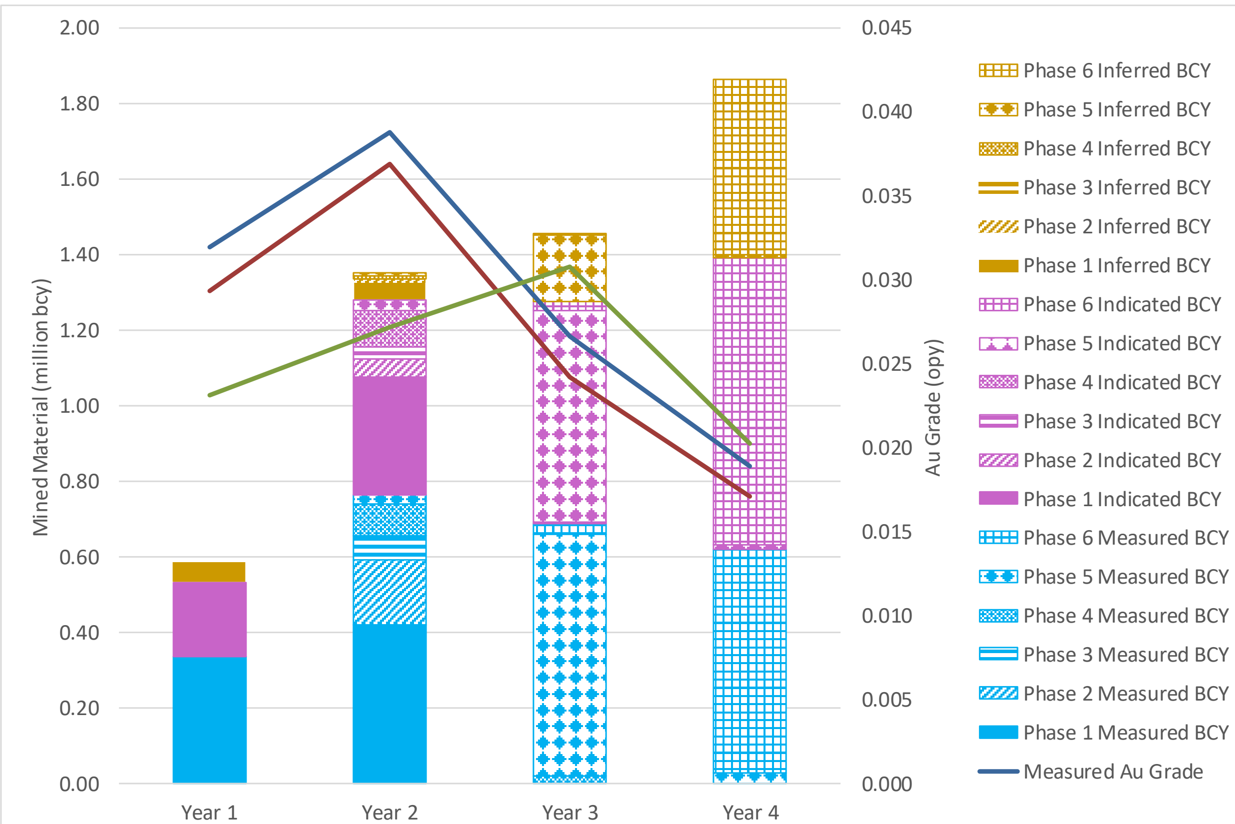
Table 16-4: Chandalar Mine Base Case Measured, Indicated, and Inferred Mined Resource Schedule

Pit	Year 1	Year 2	Year 3	Year 4	Year 5	Total
	Measured Material bcy (millions)					
Phase 1	0.33	0.42				0.75
Phase 2		0.17				0.17
Phase 3		0.06				0.06
Phase 4		0.08	0.02			0.10
Phase 5		0.03	0.64	0.02		0.69
Phase 6			0.03	0.60	0.09	0.71
Total	**0.33**	**0.76**	**0.69**	**0.62**	**0.09**	**2.49**
	Indicated Material bcy (millions)					
Phase 1	0.20	0.31				0.51
Phase 2		0.05				0.05
Phase 3		0.03				0.03
Phase 4		0.10	0.00			0.10
Phase 5		0.03	0.56	0.01		0.60
Phase 6			0.02	0.76	0.10	0.88
Total	**0.20**	**0.52**	**0.59**	**0.77**	**0.10**	**2.17**

Pit	Year 1	Year 2	Year 3	Year 4	Year 5	Total
	Inferred Material bcy (millions)					
Phase 1	0.053	0.046				0.10
Phase 2		0.01				0.01
Phase 3		0.0004				0.0004
Phase 4		0.009				0.009
Phase 5		0.006	0.17	0.01		0.188
Phase 6			0.0074	0.464	0.037	0.508
Total	**0.053**	**0.07**	**0.1813**	**0.472**	**0.037**	**0.81**
	Measured Au Oz (1000s)					
Phase 1	10.64	13.56				24.20
Phase 2		5.65				5.65
Phase 3		3.03				3.03
Phase 4		6.94	1.29			8.23
Phase 5		0.45	16.56	0.88		17.89
Phase 6			0.46	10.84	2.60	13.89
Total	**10.64**	**29.63**	**18.30**	**11.72**	**2.60**	**72.89**
	Indicated Au Oz (1000s)					
Phase 1	5.81	8.50				14.31
Phase 2		1.75				1.75
Phase 3		1.26				1.26
Phase 4		7.14	0.15			7.30
Phase 5		0.43	13.71	0.37		14.51
Phase 6			0.39	12.77	2.23	15.39
Total	**5.81**	**19.08**	**14.25**	**13.14**	**2.23**	**54.52**
	Inferred Au Oz (1000s)					
Phase 1	1.23	1.01				2.24
Phase 2		0.23				0.23
Phase 3		0.02				0.02
Phase 4		0.55				0.55
Phase 5		0.06	5.45	0.33		5.83
Phase 6			0.134	9.24	0.59	9.97
Total	**1.23**	**1.87**	**5.580**	**9.57**	**0.59**	**18.85**
	Measured Au Grade (opy)					
Phase 1	0.032	0.032				0.032
Phase 2		0.033				0.033
Phase 3		0.048				0.048
Phase 4		0.082	0.071			0.080
Phase 5		0.017	0.026	0.035		0.026
Phase 6			0.016	0.018	0.029	0.020
Total	**0.032**	**0.039**	**0.027**	**0.019**	**0.029**	**0.029**
	Indicated Au Grade (opy)					
Phase 1	0.029	0.027				0.028
Phase 2		0.038				0.038
Phase 3		0.040				0.040
Phase 4		0.072	0.055			0.072
Phase 5		0.016	0.024	0.033		0.024
Phase 6			0.016	0.017	0.023	0.017

Pit	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Total	**0.029**	**0.037**	**0.024**	**0.017**	**0.023**	**0.025**
	Inferred Au Grade (opy)					
Phase 1	0.023	0.022				0.023
Phase 2		0.029				0.029
Phase 3		0.036				0.036
Phase 4		0.064				0.064
Phase 5		0.010	0.031	0.038		0.031
Phase 6			0.018	0.020	0.016	0.020
Total	**0.023**	**0.027**	**0.031**	**0.020**	**0.016**	**0.023**

Figure 16-4: Chandalar Mine Base Case Measured, Indicated, and Inferred Mined Resource Schedule



16.6 Processing Schedule

High-grade material stockpiled during ramp up periods would be processed at the completion of mining, and low-grade stockpiled material would be processed following completion of high-grade stockpiled material processing. The processing schedule is shown in Table 16-5 and Figure 16-5.

Table 16-5: Chandalar Mine Material Processing Schedule

Pit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
	High Grade Material bcy Processed (millions)							
Phase 1	0.30	0.54	0.00	0.00	0.00	0.00	0.00	0.83
Phase 2	0.00	0.17	0.00	0.00	0.00	0.00	0.00	0.17

Pit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Phase 3	0.00	0.08	0.00	0.00	0.00	0.00	0.00	0.08
Phase 4	0.00	0.18	0.02	0.00	0.00	0.00	0.00	0.20
Phase 5	0.00	0.03	0.95	0.04	0.00	0.00	0.00	1.02
Phase 6	0.00	0.00	0.04	0.92	0.18	0.00	0.00	1.14
Total	**0.30**	**0.99**	**1.00**	**0.97**	**0.18**	**0.00**	**0.00**	**3.44**
High Grade Au Raw Ounces Processed (1000s)								
Phase 1	16.65	21.20	0.00	0.00	0.00	0.00	0.00	37.85
Phase 2	0.00	7.19	0.00	0.00	0.00	0.00	0.00	7.19
Phase 3	0.00	4.15	0.00	0.00	0.00	0.00	0.00	4.15
Phase 4	0.00	14.55	1.40	0.00	0.00	0.00	0.00	15.95
Phase 5	0.00	0.74	32.38	1.56	0.00	0.00	0.00	34.68
Phase 6	0.00	0.00	0.78	25.78	4.98	0.00	0.00	31.55
Total	**16.65**	**47.83**	**34.57**	**27.35**	**4.98**	**0.00**	**0.00**	**131.37**
Stockpiled Material Processed								
High Grade BCY (millions)	0.00	0.00	0.00	0.00	0.20	0.00	0.00	0.20
Low Grade BCY (millions)	0.00	0.00	0.00	0.00	0.58	0.96	0.34	1.89
High Grade Au Oz (1000s)	0.00	0.00	0.00	0.00	6.95	0.00	0.00	6.95
Low Grade Au Oz (1000s)	0.00	0.00	0.00	0.00	4.60	7.58	2.71	14.89

Figure 16-5: Chandalar Mine Material Processing Schedule

16.7 Mine Operation and Layout

A final reclamation plan was developed that provides for partial backfilling of the pits. As much as possible, waste material will be backfilled into the pits concurrent with mining activities.

The mine layout would include a process wash site, high-grade and low-grade mineralized material stockpiles, interim waste dumps required to store waste that cannot be directly backfilled into the pits, and final waste dumps for material not scheduled to be returned to the pits.

The process material and waste would be drilled and blasted using a rotary crawl driller and ammonium nitrate fuel oil (ANFO). Process material would be hauled using dump trucks from the pit to the wash site, while waste rock would be hauled using dump trucks to the interim waste dumps or previously mined pits.

16.8 Mine Equipment Productivity

Ms. Lane estimated cycle times and determined the equipment size and numbers of trucks and loaders that would be required to meet the project schedule. A simplified approach to cycle calculations was used; it considered productivity variables such as average daily production of process material and waste, average truck haul distance and travel speed, hours per shift and shifts per day, availability variables such as breaks during the day, and truck and loader/shovel capacities. Hourly production rates and truck and loader wait times were calculated to optimize the design. The final analysis uses Caterpillar 745G trucks, with a heaped capacity of 32.7 loose cubic yards, and Caterpillar 982M loaders, with a bucket capacity of 5 loose cubic yards.

16.9 Mine Haul Roads

Surface and in-pit haul roads are designed with a width of 45 feet. The maximum road grade in pit is 10%.

16.10 Bench Height

For scheduling, Ms. Lane used a 20-foot bench height with a 68.2-degree batter angle for the Chandalar Mine pit designs. In practice, mining a shorter bench height may be preferable to increase selectivity. Catch benches would occur on every bench with a width of 12 feet. This results in overall inter-ramp pit slopes of 45 degrees.

16.11 Access Road Width

Access roads are designed with a width of 45 feet to accommodate the proposed equipment fleet, including ditches and berms. The access road would be wide enough to accommodate two-way traffic. The maximum road gradient is 10%.

16.12 Waste Rock Dump

Three waste rock dumps (WRDs) are planned at different locations around the pits. The WRDs are designed to contain total of 8.1 million loose cubic yards of waste material. The sides of the WRD are at a 2.5:1 slope, and the material would be end dumped. The remaining waste will be placed as concurrent backfills in mined out phases of the pits. Thirty-five (35) % of the waste material is concurrently backfilled in the mining pits. The backfilling of the previously mined out pits during the active mine life is planned to minimize the amount of waste material that needs to be reclaimed at the end of the mining operation. The backfills are designed at a slope of 3:1 (h:v) suitable for the reclamation plans for closure as per BLM and ACOE requirements. The pits would be backfilled from the bottom up to meet the reclamation plan. The backfill and WRDs are utilized concurrently through the mine life for waste produced from the pit.

After mining has finished, the pits will be filled with the waste material from the WRDs to meet the reclamation plans.

16.13 Mine WRD Development Schedule

Pre-stripping and production bench waste at the start of the schedule are stored in the WRDs. The waste from the lower pits is stored on the dump north of the lower pit (Dump 1). The waste from the upper pits is stored in Dump 2 and Dump 3, on the west and east banks, respectively. Once a pit Phase is complete, it is available, and backfilling can begin concurrent with production mining. When Phase 1 backfill is full in Year 1, waste storage resumes in the WRD north of the pit until later in Year 2 when pit Phase 2 is available for backfill. Phase 2 is backfilled until it is filled in Year 3 and Year 4, the final year of mining. The remaining waste in Year 4 is placed into the open Phase 3 backfill. Table 16-6 shows the waste movement during the production and reclamation periods. After reclamation is completed, 5.9 million cubic yards of waste material will remain in the WRDs.

Table 16-6: Chandalar Mine Waste Movement During Mine Production Periods (1000s bcy)

WRD	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Dump 1	924	2,075				**2,999**
Dump 2	1,180	1,821				**3,001**
Dump 3		1,283		807		**2,090**
Phase 1 BF			2,207			**2,207**
Phase 2 BF			1,298			**1,298**
Phase 3 BF			95	180		**275**
Phase 4 BF			177			**177**
Phase 5 BF				320	118	**438**
Phase 6 BF						**0**

Table 16-7: Chandalar Mine Waste Movement During Reclamation Period (1000s bcy)

WRD	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Phase 1 BF							**0**
Phase 2 BF							**0**
Phase 3 BF							**0**
Phase 4 BF							**0**
Phase 5 BF					125		**125**
Phase 6 BF					716	1,346	**2,062**

16.14 Interim Mine Plans

Figure 16-6 through Figure 16-14 illustrate the project progress at the end of each year or interim milestone of activity.

Figure 16-6: Chandalar Mine Year 1 Interim Mine Plan

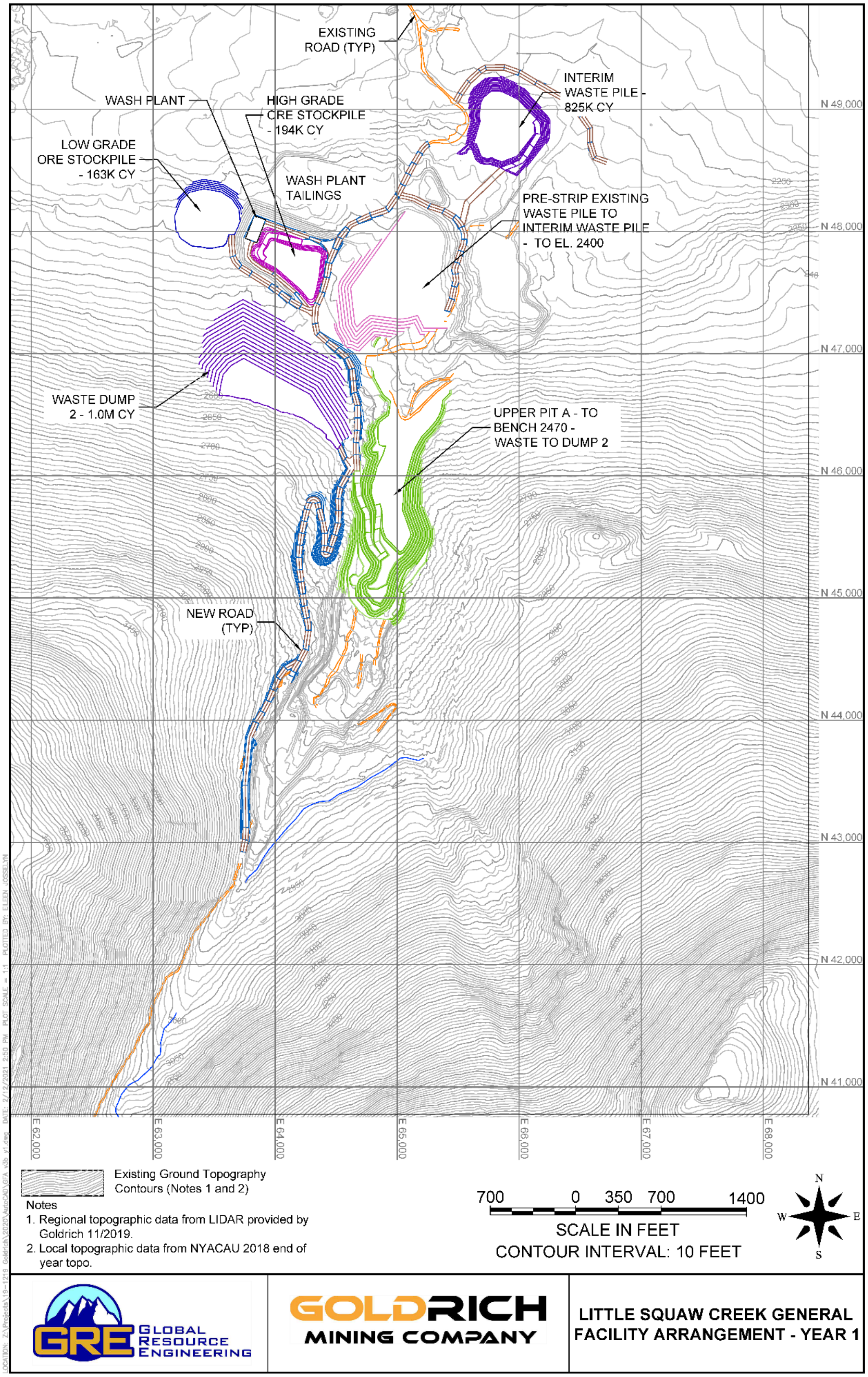


Figure 16-7: Chandalar Mine Year 2-1 Interim Mine Plan

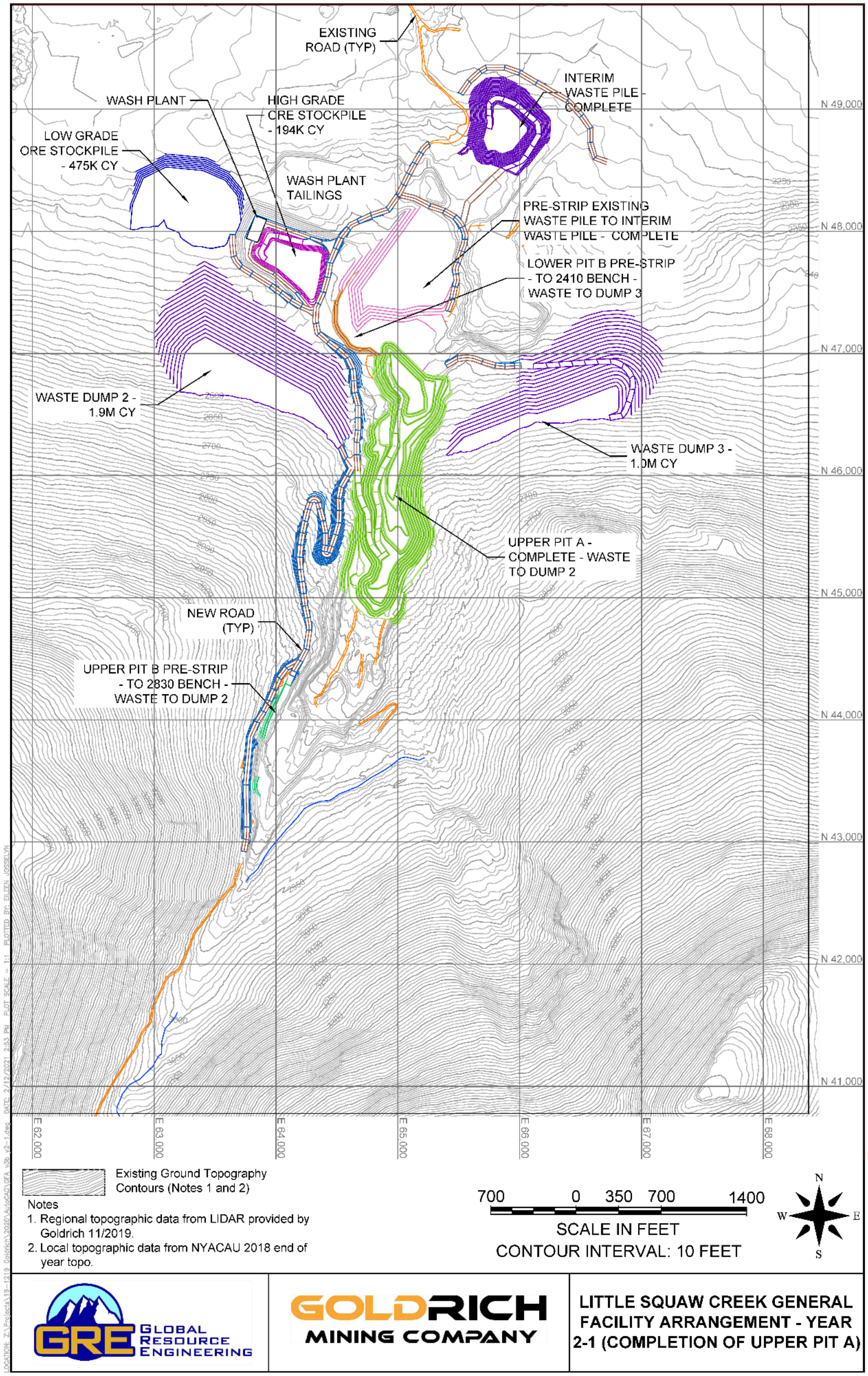


Figure 16-8: Chandalar Mine Year 2-2 Interim Mine Plan

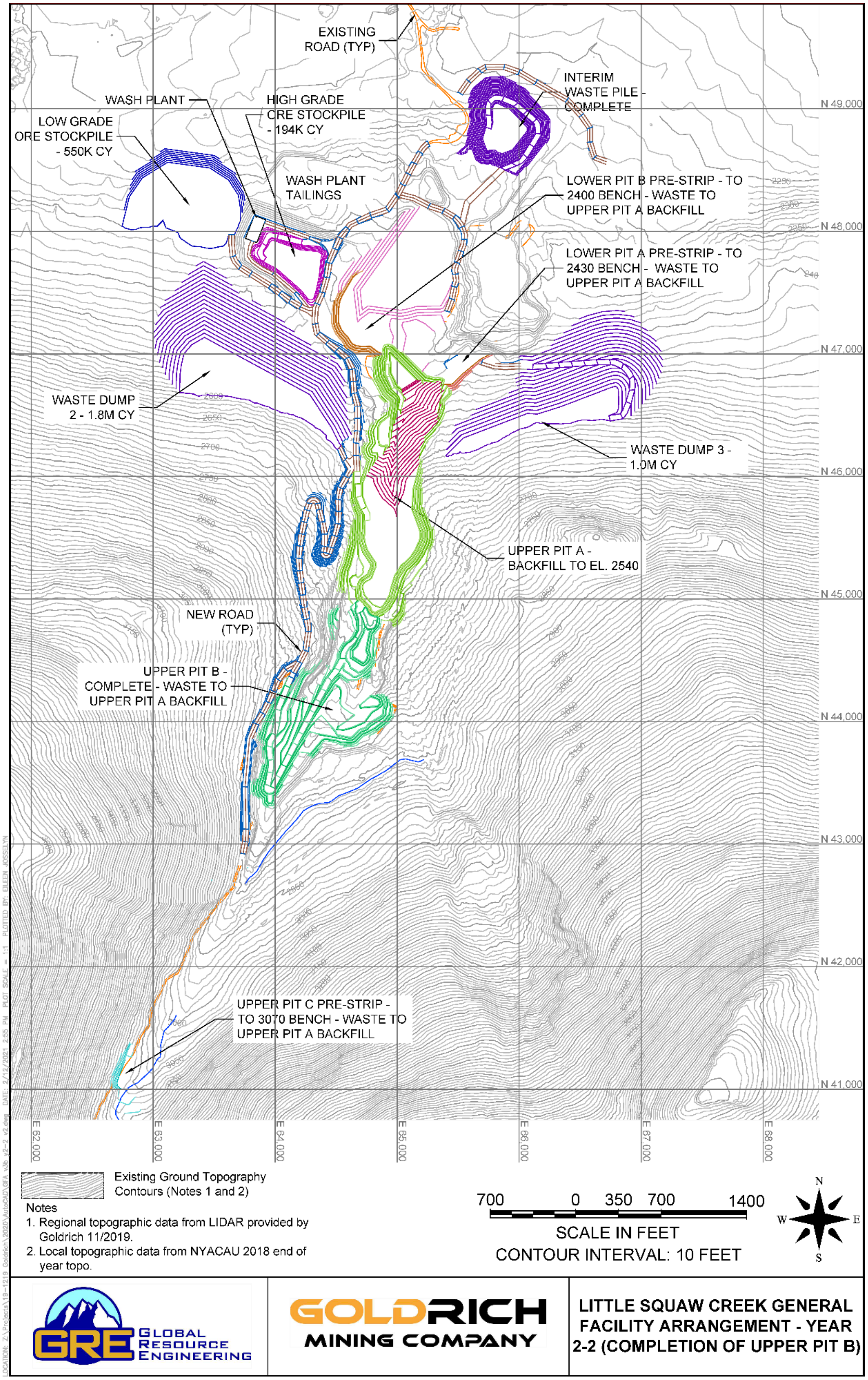


Figure 16-9: Chandalar Mine Year 2-3 Interim Mine Plan

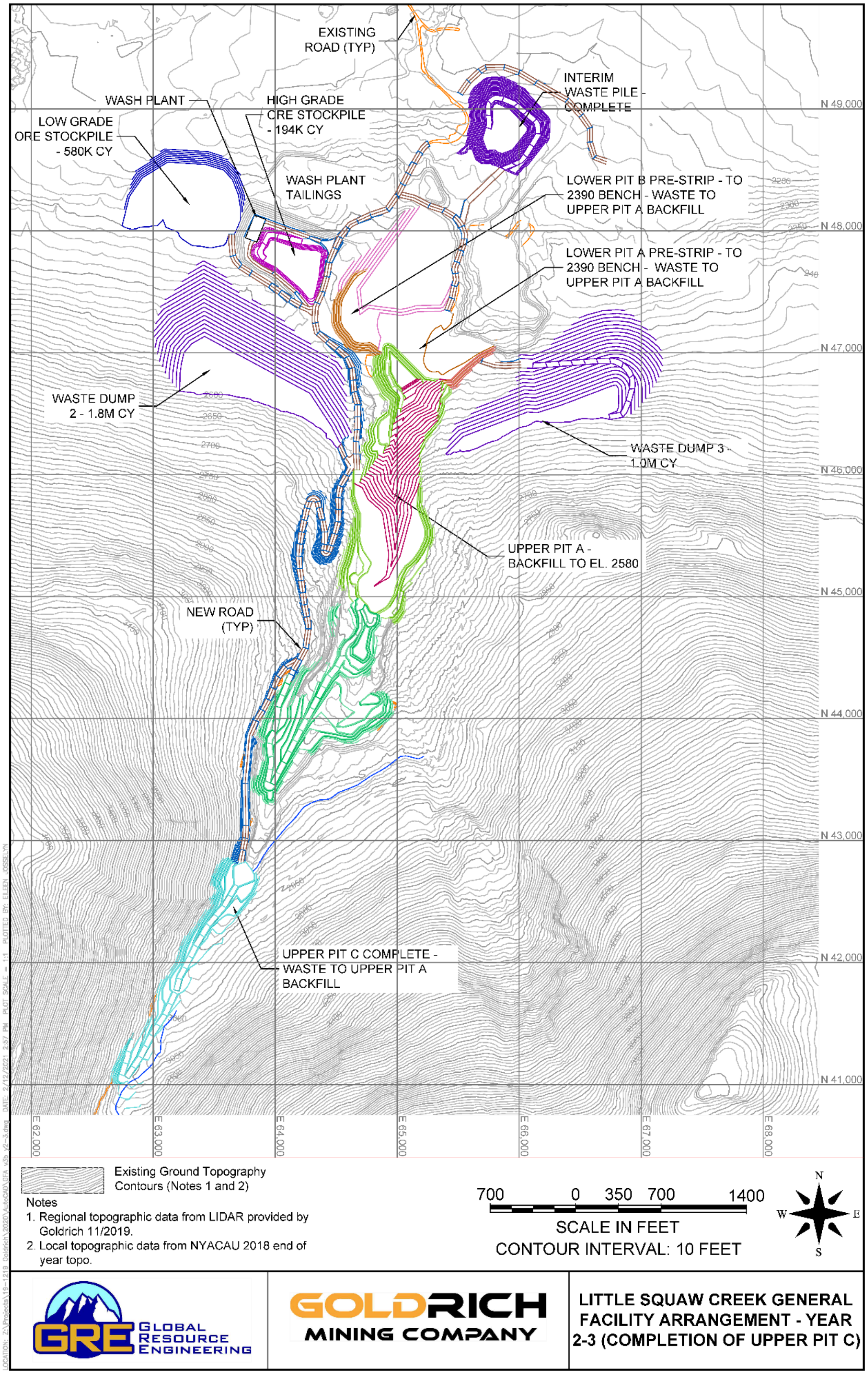


Figure 16-10: Chandalar Mine Year 2-4 Interim Mine Plan

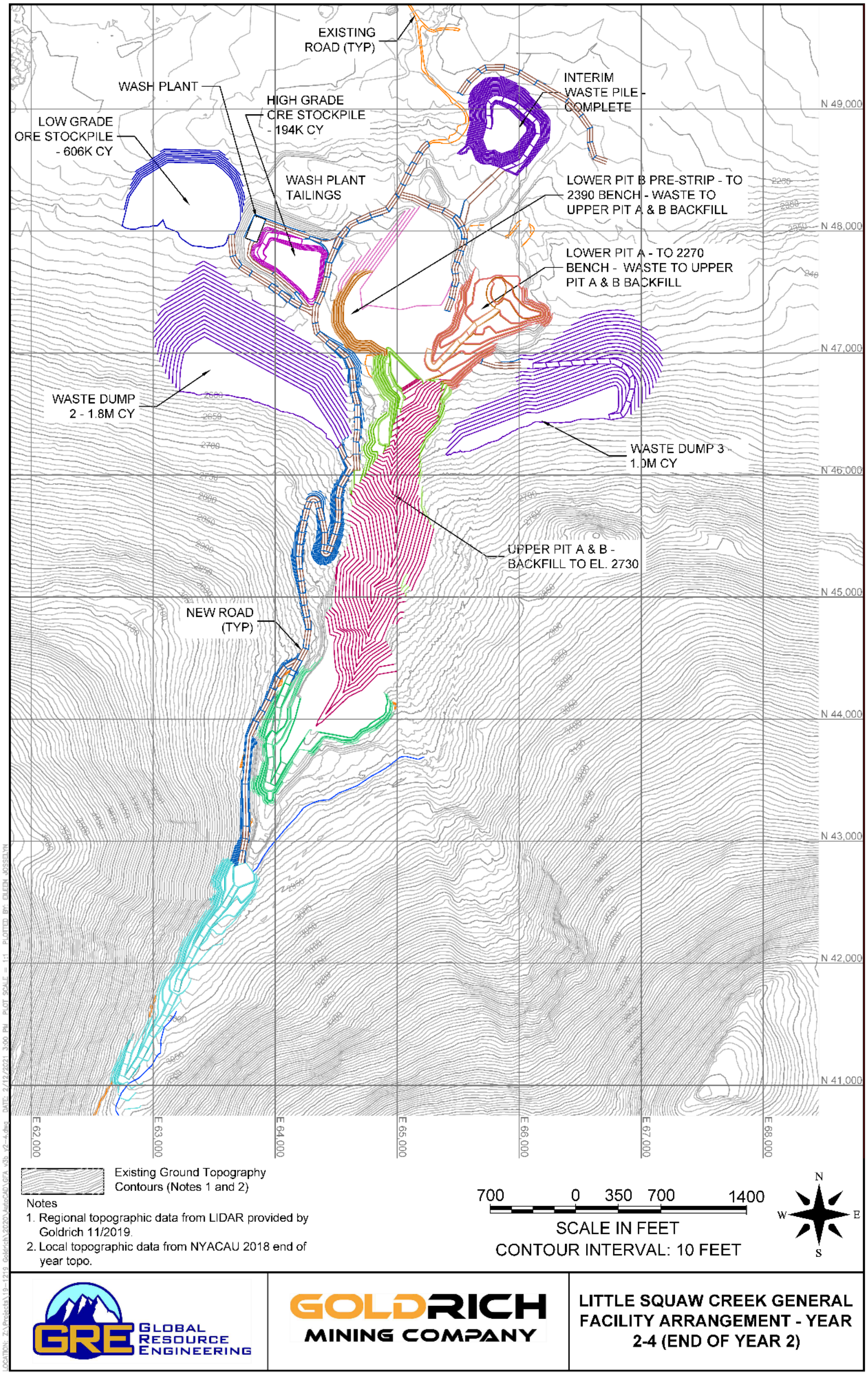


Figure 16-11: Chandalar Mine Year 3 Interim Mine Plan

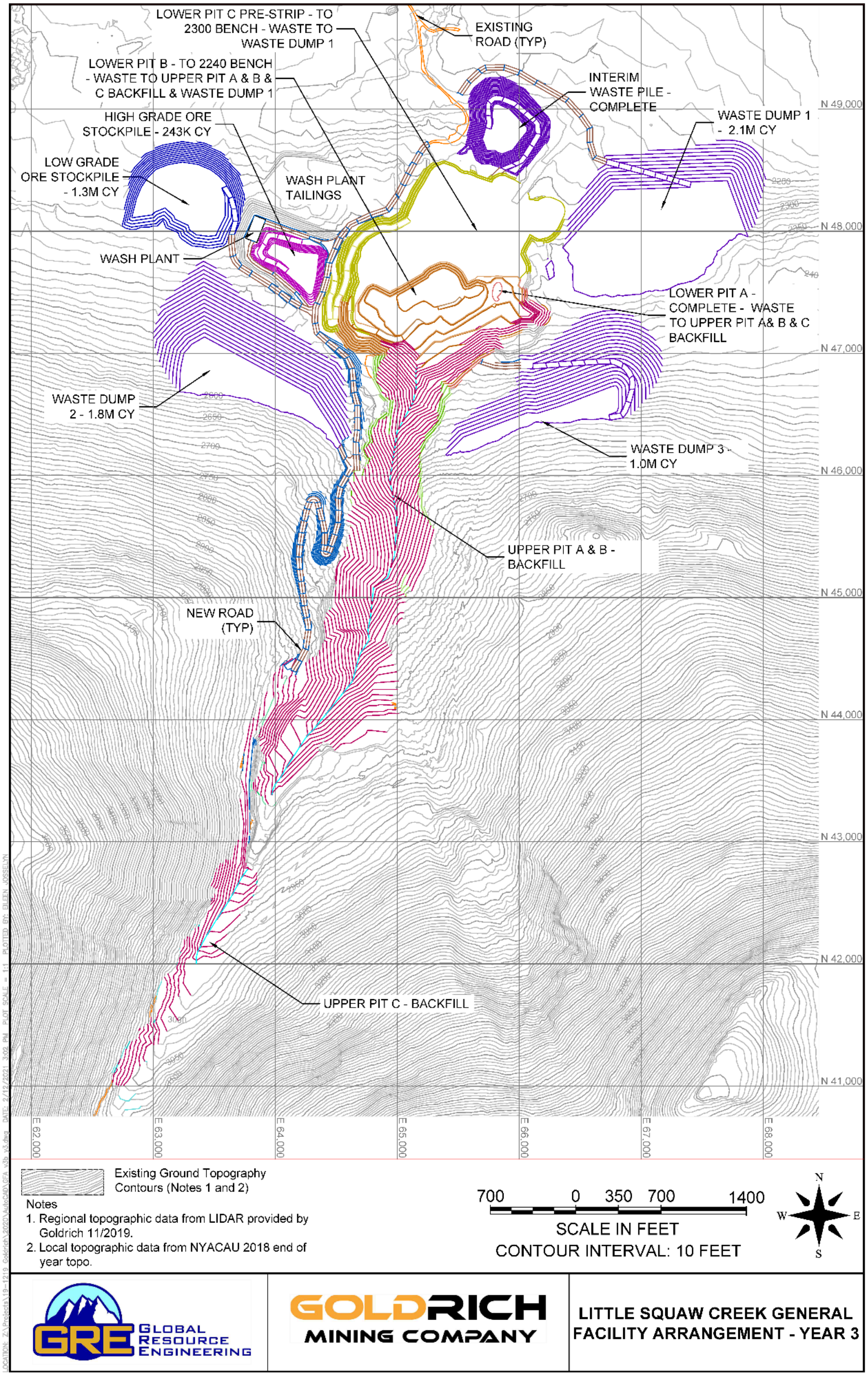


Figure 16-12: Chandalar Mine Year 4 Interim Mine Plan

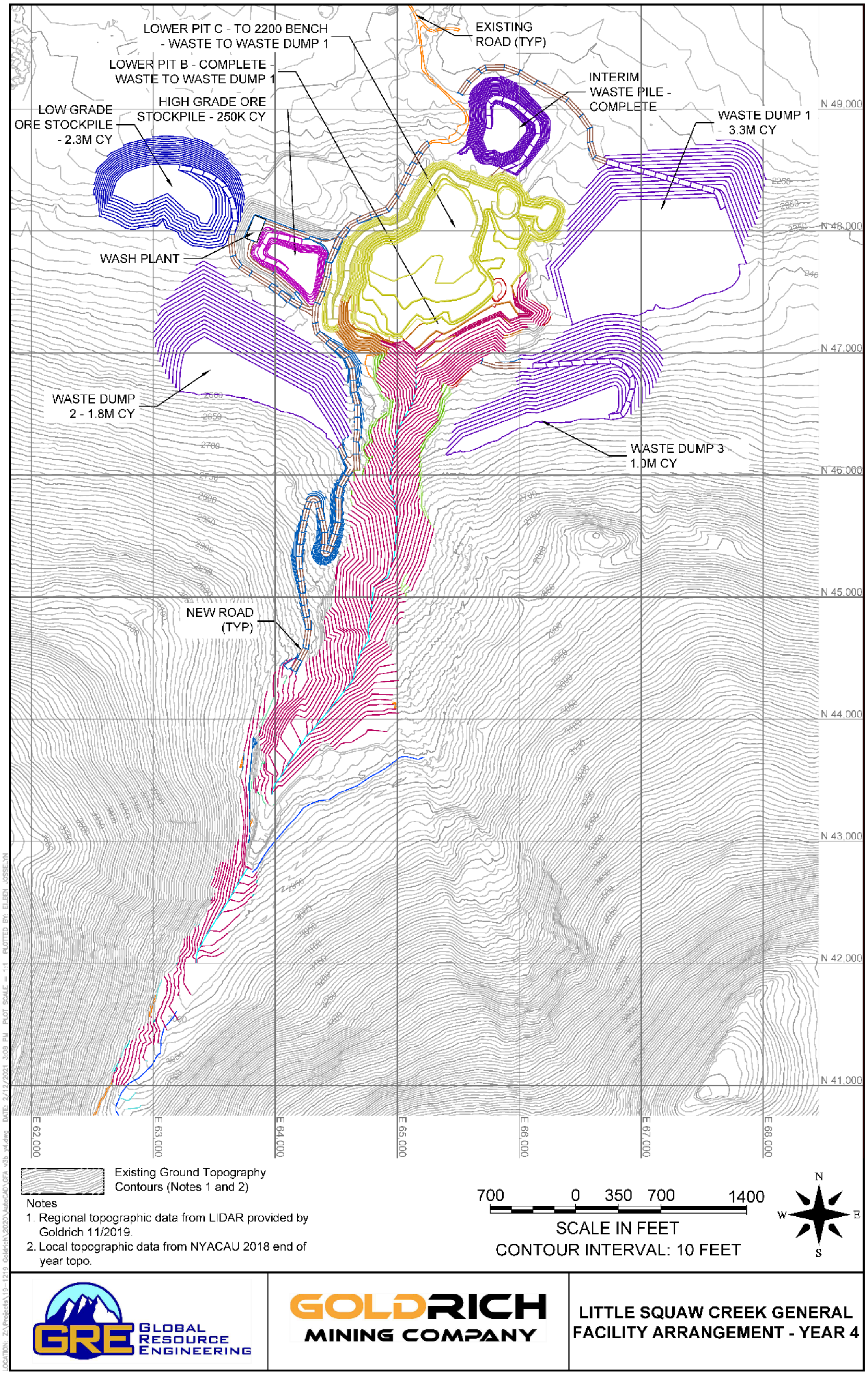


Figure 16-13: Chandalar Mine Year 5 Interim Mine Plan

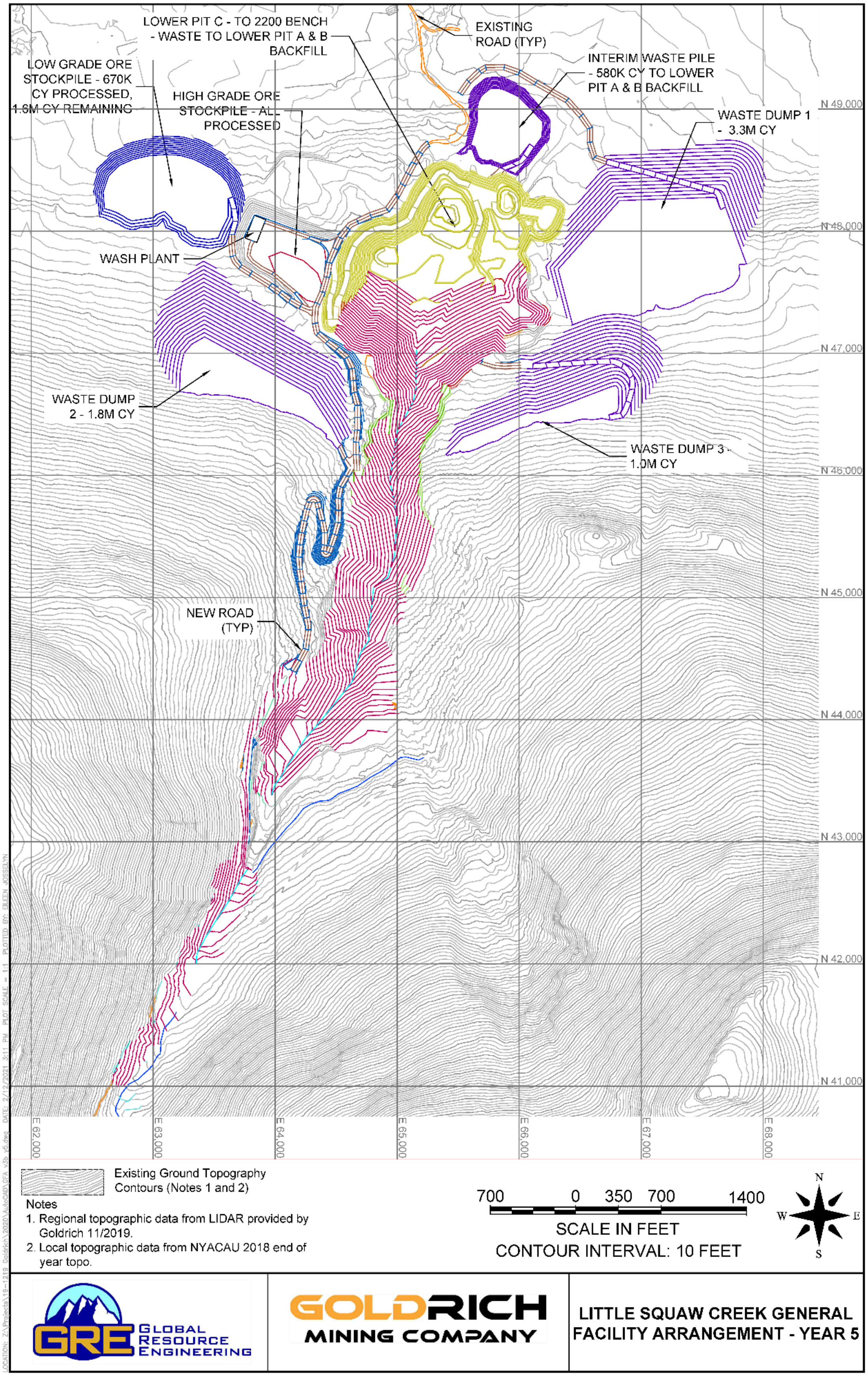
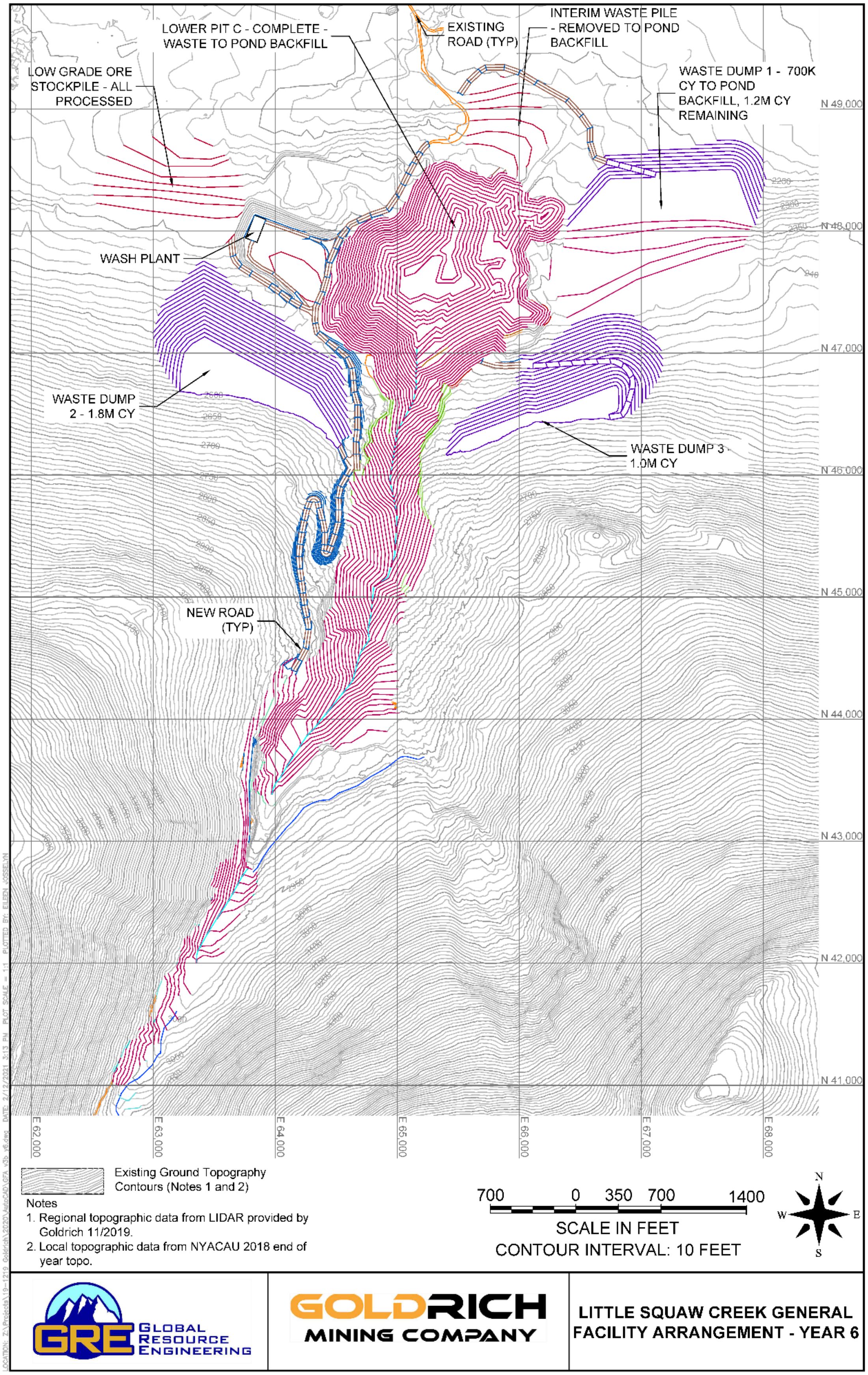


Figure 16-14: Chandalar Mine Year 6 Final Mine Plan



17.0 Recovery Methods

Prior to 2013, the historical recovery methods at Chandalar Mine have used gravity systems consisting of a rotating trommel and sluice arrangement, similar to the historic unit one shown in Photo 17-1.

Photo 17-1: Typical Trommel Gold Recovery System

For the above wash plant, the material is transported by a conveyor to a feed hopper or directly loaded into the feed hopper by an excavator or front-end loader. Water is used to transport the material from the feed hopper into a rotating trommel. The trommel is equipped with screen panels, allowing the undersize containing the gold to be carried out of the trommel to sluice boxes for recovery while the oversize is transported out of the trommel as coarse tailings. The selection of an appropriate gravity recovery method is often one of preference, but Chandalar Mine contains significant clay, making the use of a trommel less than ideal.

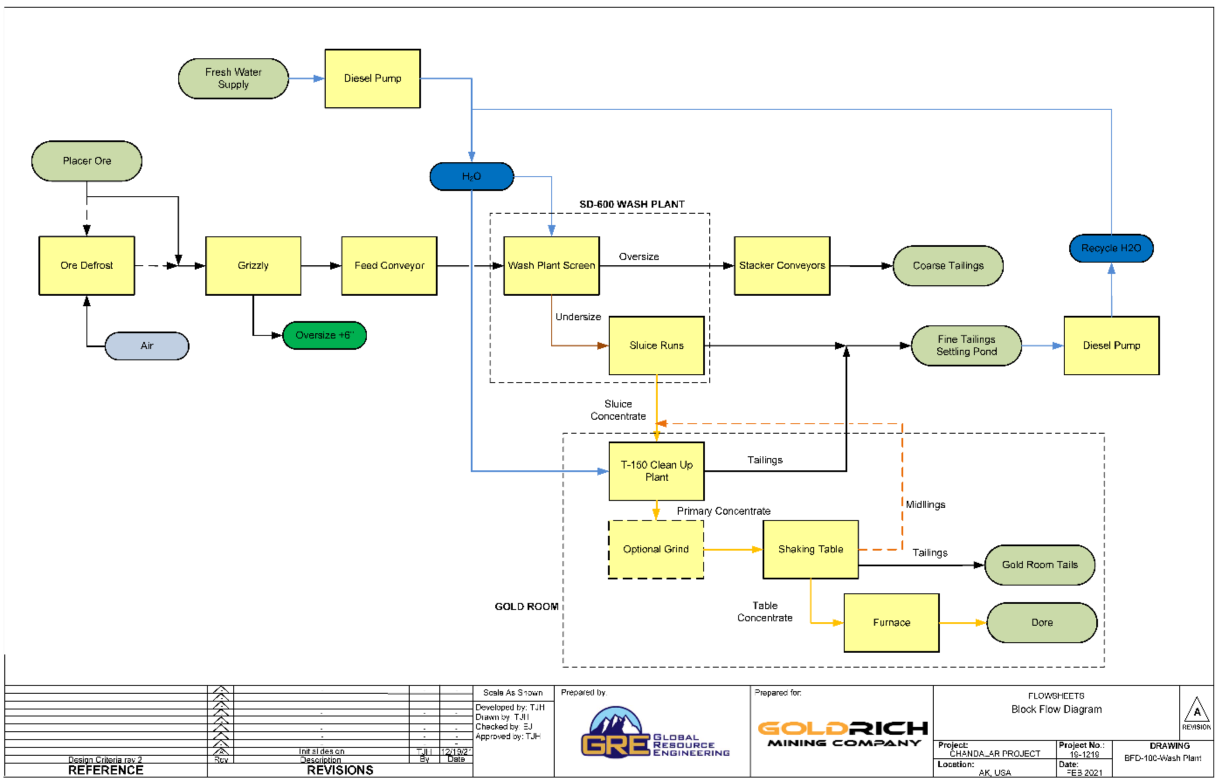
Many people believe that trommels are better suited for clay, but this is as compared to older conventional shaker screens or de-rockers. Newer shaker deck systems use high amplitude screens and directed water sprays to effectively wash the gold particles off the coarse rock with large amounts of clay present. These systems use less water and produce a cleaner coarse tailing and thus have a higher gold recovery potential. A wash plant using the newer shaker deck system was used at Chandalar Mine for the 2013 through 2018 production seasons. It is for these reasons that Dr. Harvey has selected a vibrating screen shaker system for the Little Squaw project.

17.1 Process Plant Description

Dr. Harvey has extensive experience with placer recovery equipment and as a result has selected a state-of-the-art system for the Chandalar Mine project. Further, the rehandling of materials and haulage are two of the major expenses associated with placer mining, and a proper design can significantly reduce these costs. As a result of this, Dr. Harvey has recommended the use of two mobile wash plant shaker systems combined with a feed hopper with a side dumping grizzly, a belt feeder, and plant feed conveyor. The plant consists of a feed receiver with water sprays, a vibrating screen deck, and a double sluice box.

The coarse tailings are transported via a series of portable conveyors to a slewing stacker. Having two separate plants that can be located close to the active mining area reduces haulage and maximizes operating time through redundancy. Further, having a feed hopper with a grizzly reduces oversize damage to the screen and hopper damage from the mobile equipment. The use of portable conveyors and a stacking conveyor reduce the rehandling of coarse tailings and allow strategic placement in a single handle, reducing rehandle and closure costs. Figure 17-1 shows the conceptual process flow sheet for the project.

Figure 17-1: Chandalar Mine Process Flow Diagram



17.1.1 Wash Plant

The SD600 comes in both the "S" and "M" versions. The "M" version is designed to be more mobile than the "S" model, mainly through the addition of heavy-duty skids and retractable sluice boxes. Both machines can be relocated, but the "M" version is designed to be moved with less effort. Photo 17-3 shows a schematic of the SD600m.

The SD600 is one of the largest wash plants available, and it has a proven track record in rugged environments and sitting through harsh winters. The unit can be fully containerized for transport to site via barge, winter haul road, or plane. The plant is designed to process 200-250 cubic yards per hour, with a target product rate of 4,000 bcy per day based on 20 operating hours per day. The unit uses between 2,500 and 3,500 gpm of water depending on the tonnage and clay content and is designed to use heavily silted recycle water typical of placer operations. The unit has two main motors, including a 40 horsepower

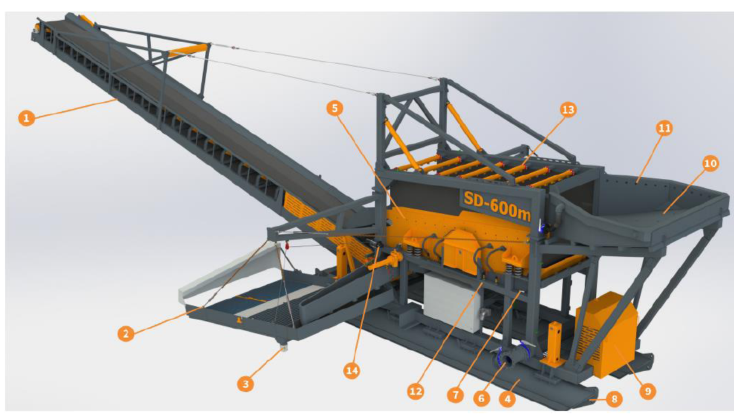
Photo 17-2 shows the recommended gravity recovery system.

The SD600 comes in both the “S” and “M” versions. The “M” version is designed to be more mobile than the “S” model, mainly through the addition of heavy-duty skids and retractable sluice boxes. Both machines can be relocated, but the “M” version is designed to be moved with less effort. Photo 17-3 shows a schematic of the SD600m.

The SD600 is one of the largest wash plants available, and it has a proven track record in rugged environments and sitting through harsh winters. The unit can be fully containerized for transport to site via barge, winter haul road, or plane. The plant is designed to process 200-250 cubic yards per hour, with a target product rate of 4,000 bcy per day based on 20 operating hours per day. The unit uses between 2,500 and 3,500 gpm of water depending on the tonnage and clay content and is designed to use heavily silted recycle water typical of placer operations. The unit has two main motors, including a 40 horsepower

Photo 17-2: MACON – SD600m “Slucifer” Gravity Recovery System

Photo 17-3: SD600m Schematic



1 53" X 42' Conveyor
2 Fold up tables for plant moves
3 Boil box cleanouts
4 Skid bottom wear plates
5 6' x 14' double shaker deck
6 10" water inlet
7 Catch tray doubles as primary sluice box
8 Skid tip tow lugs
9 Plant mounted power pack
10 Hopper wear liners
11 Interior hopper spray bars
12 Waterworks built into framework
13 7x top deck spray bars
14 All 3 tables hinged by hyd. winches

(hp) screen deck motor and a 12.5 hp conveyor motor. The units can be equipped with an onboard generator or supplied power from an external source.

17.1.2 Feed System

The feed system consists of the following equipment:

- A 42-inch mobile heavy-duty belt feeder assembly complete with 18-inch main frame chassis and a fifth wheel pin assembly, including 8 tires, air brakes, running lights and mud flaps and four blocking legs.
- The hopper is 16-feet x 8-feet, having a capacity of 20 cubic yards and equipped with a self-relieving bottom opening and 0.25 plate liners. The hopper is equipped with a hydraulic side dumping grizzly with tri-bar adjustable bars with 2-inch adjustments. The hydraulics use 4-inch diameter heavy cylinders and a remote control to allow operation from the loading equipment cabin. The unit has a 15 hp electric motor operating an 18 gpm hydraulic pump.
- The main feed belt is a 42-inch x 20-foot removable belt feeder cartridge using 3 ply-600 belting equipped with a 10 hp variable speed motor with remote control.
- The main discharge conveyor is 48 inches x 46 feet with a 20 hp electric motor.

17.1.3 Coarse Tailings

The coarse tailings from the wash plant report to a series of portable or "jump" conveyors. These conveyors are 100 feet long with a 36-inch wide belt and are equipped with a 30 hp electric motor. The jump conveyors transport the coarse tailings to a radial stacker. The stacker is 100 feet long with a 36-inch belt and is capable of slewing 270 degrees, producing a stockpile of approximately 28,600 cubic yards or roughly a week's production. The stacker belt is driven by a 30 hp electric motor, and the slewing mechanism is driven by a hydraulic pump operated by a 5 hp electric motor.

17.1.4 Fine Tailings

Normally, the fine tailings discharged from the sluice are allowed to enter a nearby tailings impoundment where the solids settle and the water is recovered. In this case, Dr. Harvey has recommended that the fine tailings be pumped to a disused section of the mine to allow the tailings to act as backfill and allow the water to still be recovered. The tailings will ideally be placed uphill from the plant and the water collected back at the plant site for reuse. This reduces the need to establish new ponds after each wash plant move and reduces the rehandling of the fine tailings for closure.

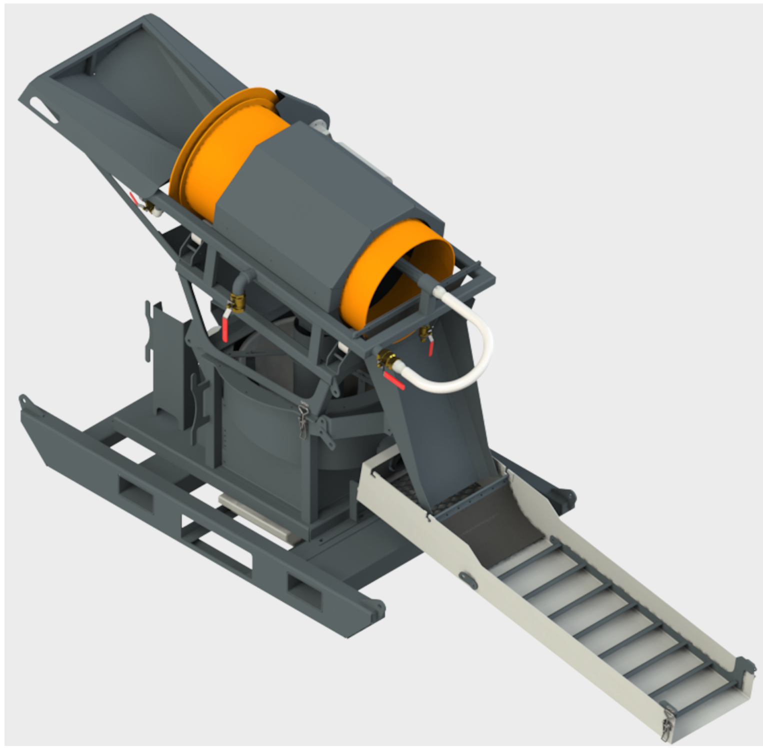
17.1.5 Water Pumps

Each wash plant requires significant water flow, and an 8-inch pump with 10-inch suction has been selected. This pump is a John Deere diesel powered Cornell pump with an engine rating of 225 continuous hp. Capacity is 4,000 gpm at 250 feet of head. The pump has a welded structural steel base with lift point, radiator guard, and battery mount.

17.1.6 Concentrate Clean Up

The concentrate recovered from the wash plants typically requires further cleaning before selling to a refinery. Dr. Harvey recommends the T-150 clean up plant coupled with a Gemini GT250 shaking table. The T-150 (Photo 17-4) is basically a mini wash plant designed to allow rapid upgrading of the concentrate without significant material handling issues. This unit can process three to four cubic meters per hour through an 18-inch trommel drum with 0.25-inch screen. The unit has a Knudson bowl with bevel gear drive and rubber liner and a sluice run for both coarse tailings and Knudson bowl tailings. The system is skid mounted, complete with a 2-inch water pump with suction line and 50 feet of layflat hose.

Photo 17-4: T-150 Clean Up Plant



17.2 Unique Issues Related to Placer Mining

The Chandalar Mine property is located in Alaska and, therefore, the mining period is limited and much of the ground has permafrost. Historic mining has occurred from mid-May to mid-October with some variance depending on the weather. The previous operator was averaging 114 days per season or approximately 16 weeks of mining. A similar budget has been used for this project.

The existence of permafrost presents many challenges. It makes the mining difficult due to the frozen nature of the material. It turns into mud when it melts and also swells. From a processing perspective, frozen gravels do not release their gold and need to be fully thawed before treatment. Traditionally, thawing occurs in the field by removing the overburden and allowing the sun to thaw the material. This method has been exploited successfully by many operations, but it requires a lot of pre-stripping to be done in advance of mining, which is costly. Dr. Harvey has examined a series of additional concepts to help overcome the frozen ground issue, including the use of an aerated pad near the plant, solar water heating, and a steam system.

The air thawing system makes use of our knowledge of heap leaching using an on-off pad system. A permeable gravel layer acts as the pad base and contains simple drilled plastic pipes. These pipes are connected to low pressure blowers that push large volumes of warm ambient air through the material. This system involves re-handling the materials but would be designed as a method to allow the operator to get started earlier in the season and to help balance out-of-sequence stripping. This system could be augmented with steam injection to increase the air temperature or the use of solar heated water. Solar heated water can also be employed to irrigate strip ground to speed the thawing process. Dr. Harvey recommends the use of cheap durable scalable pool heaters such as those shown in Photo 17-5.

Photo 17-5: DHPE Solar Water Heater

18.0 Project Infrastructure

Goldrich has significantly improved or repaired the infrastructure at Chandalar Mine, including establishing a field camp for up to 25 people with limited shop and repair equipment, conex units for secure storage, and office facilities. The Company has repaired most of the 27.5 miles of mine road that connect all the major prospects to airstrips at Squaw Lake (5,000 feet length) and Big Creek (approximately 1,500 feet length), and the old Tobin Creek camp and mill.

There is no electrical power grid in northern Alaska. Previous mining by lessees to Goldrich have relied on diesel powered generators.

A natural spring, located on the patented 5-acre (2-ha) mill site, is reported to flow 140 gpm year-round at a temperature of 40°F (4.4°C). This parcel adjoins the Squaw Lake airstrip and is available as a future permanent camp site.

19.0 Market Studies and Contracts

Production data for Chandalar Mine between 2015 and 2018, when GNP mined the property, is available and considered an accurate representation. Gold production was reported in both raw (concentrate) and fine gold recovered. Historically, the purity value (fine gold to raw gold ratio) has been 0.844. GRE used a 3.6% deduction for refinery charges and metal deductions.

Sales of placer gold is common; market studies are not needed. A 24 month trailing average gold price of $1,650 was used in this study.Environmental Studies, Permitting and Social or Community Impact

The prior operators of the Chandalar placer mine have all the required permits needed to operate the mine. Either the mine will be required to obtain new permits to operate, or it would need to acquire the existing operating permits from NyacAU.

The Chandalar Mine has been permitted by the ACOE for reclamation work required to return the property to pre-mining conditions after completion of mining activity. This permit will require modification once Goldrich has presented the new reclamation plan to the proper authorities.

19.1 Prior ACOE Permits

Permit number POA-2009-366, Little Squaw Creek, was issued to NyacAU Mining on June 24, 2013, to discharge 1,280,000 cubic yards of overburden and rock into 120 acres of wetlands and 128 acres of uplands to construct, operate, and reclaim a mining operation at Chandalar Mine in the Chandalar Mining District.

On November 3, 2016, the operation was moved into uplands, located in the headwaters of Little Squaw Creek, and the permit was modified to authorize to construct and reclaim a 3,500 linear foot stream diversion of Little Squaw Creek.

On October 17, 2017, the permit was modified to discharge 19,200 cubic yards of gravel fill into 2.87 acres of wetlands to construct an access road and a stream crossing of Big Squaw Creek.

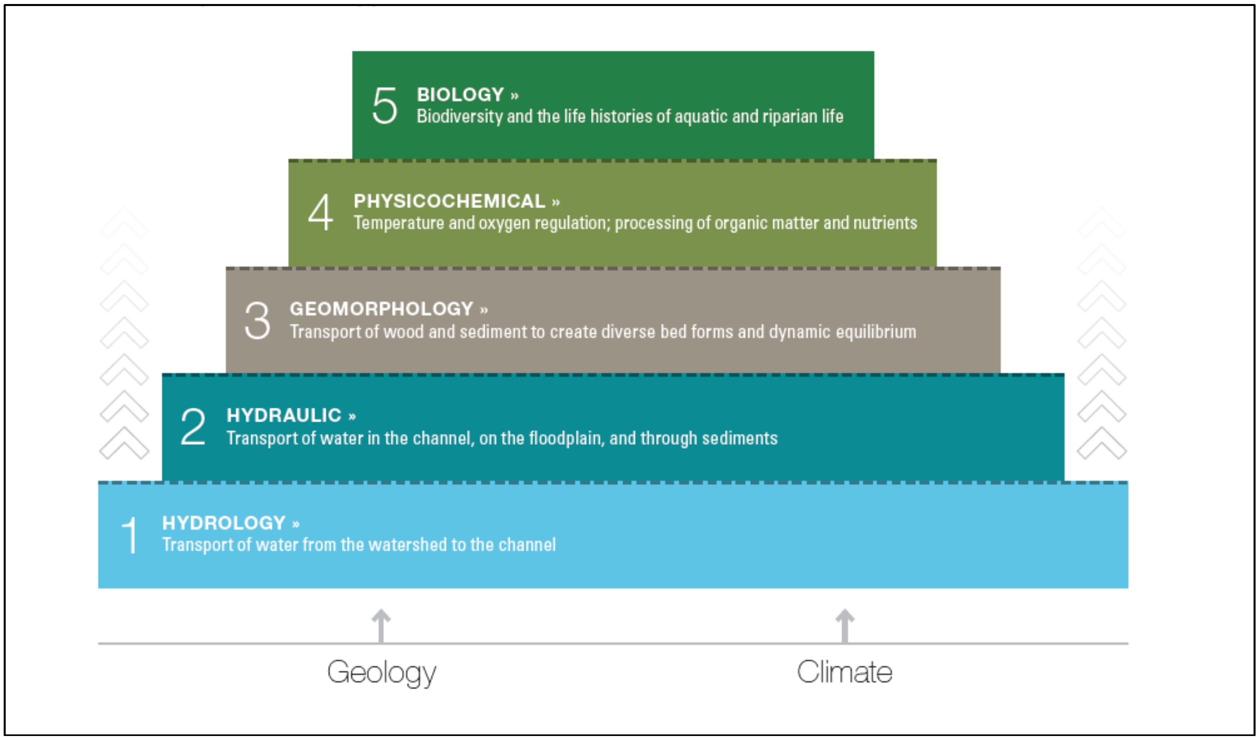
The third modification of the original permit was issued on April 8, 2019, and was modified as follows: the permittee will conduct reclamation on the Upper and Lower Mine Pits, discharging approximately 2,700,000 cubic yards of overburden material into mine pits that were excavated in wetlands and uplands, re-contouring the landscape, and restoring approximately 6,070 linear feet of stream channel. The time limit for completing the work authorized ends on April 30, 2024; however, request for a time extension may be submitted up to one month before permit expiration.

Due to the larger pit, a new permit must be acquired from the ACOE.

19.1.1 Reclaiming the Stream Channel

Reclaiming the stream channel is the primary requirement of the ACOE permit. It is important to note that the streams in the vicinity of the Project have been designated as Type II streams. Type II streams are the second-lowest classification of streams in the ACOE hierarchy, as shown in Figure 19-1.

Figure 19-1: Stream Hierarchy



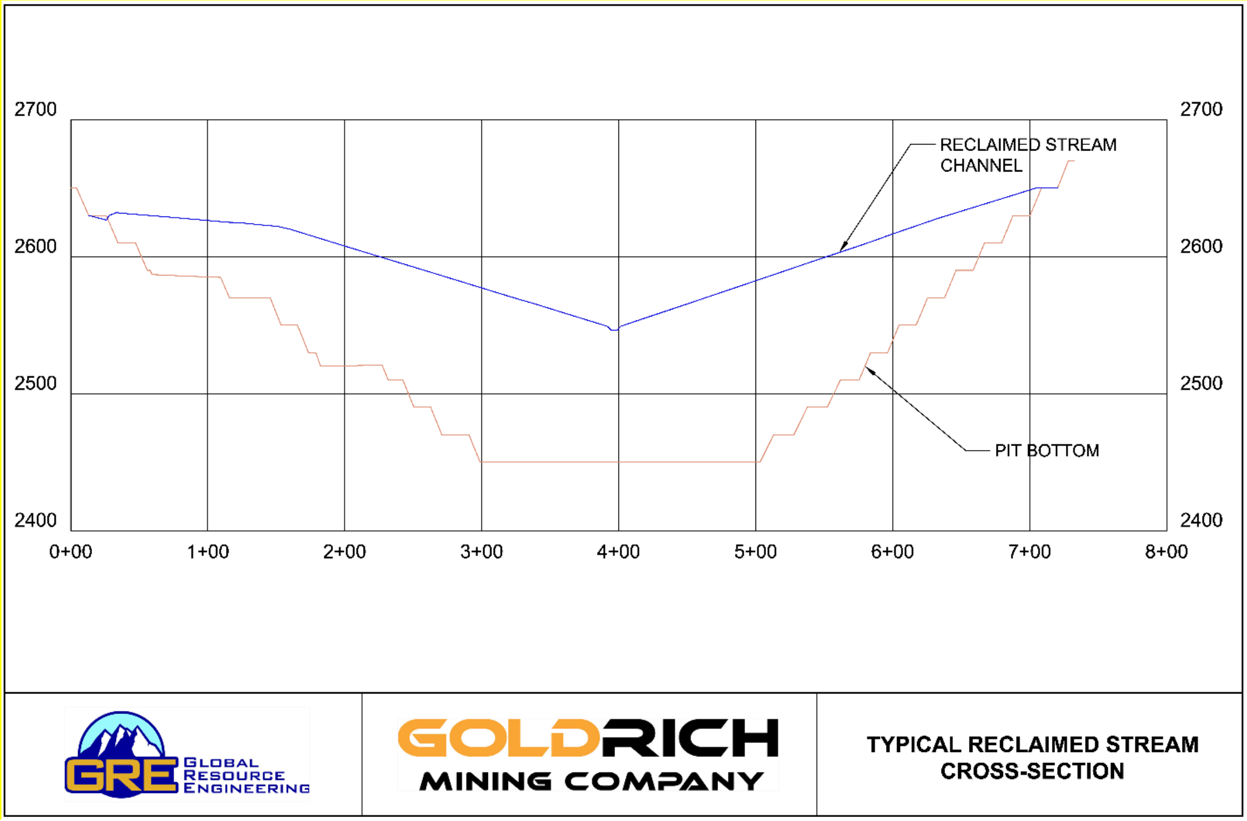
from (EPA, 2012)

One can see that the highest-level stream is expected to contain aquatic and riparian biodiversity. A level four stream is required to have temperature, oxygen, and organic matter cycling. Level three streams have dynamic sediment geomorphology. Level 2 streams are only required to transport water and sediments. Finally, Level 1 streams are required only to transport water. As a result, the reclamation of the streambeds in the project vicinity are ONLY required to transport water and sediments. They are not required to manage nutrients, temperature, nor contain aquatic life. This is a low standard for stream reclamation compared to other riparian ecosystems impacted by placer mining.

Based on the new mine plan, it will be necessary to modify the reclamation plan while still meeting the ACOE's requirements for stream channel reclamation and grading.

The new plan reclaims 7,140 feet of stream channel. The BLM and ACOE require a 3:1 slope transverse to the direction of flow, and a 3% to 10% grade longitudinal to flow. Figure 20-2 shows an example profile of the reclaimed stream channel compared to the original pit extents.

Figure 19-2: Typical Reclaimed Stream Transverse Profile



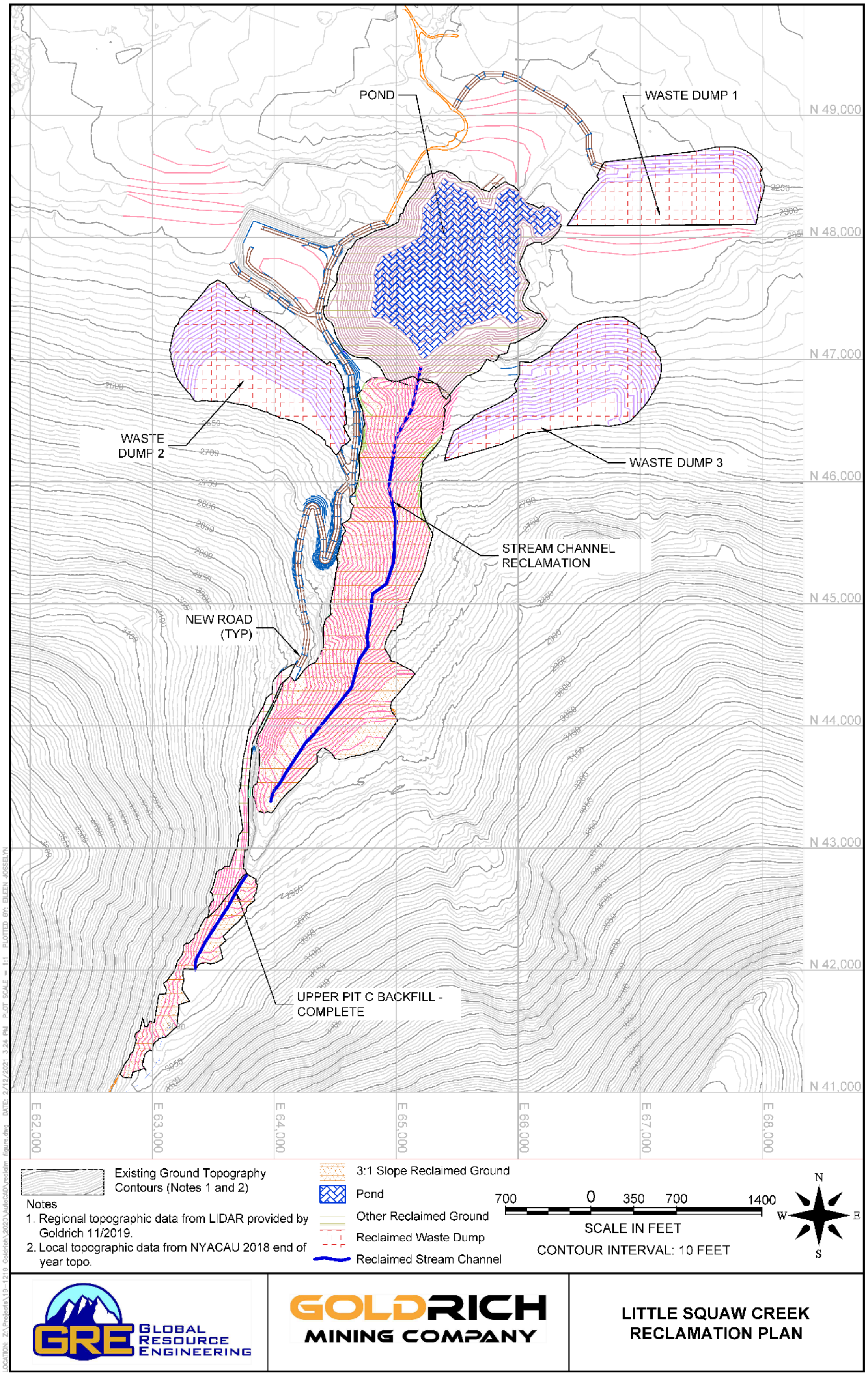
This profile maintains the sediment and water transport mechanisms of the stream channel, prevents erosion of the banks of the stream, allows for the establishment of riparian vegetation, and allows for safe access for wildlife. The area of reclaimed ground around the stream is 2.86 million square feet.

The bottom of the stream must have swale structures in order to create a natural variety of sediment deposit areas. This work is on a micro-scale compared to the larger backfilling effort and will be created in the field as per ACOE requirements. Extra equipment time has been included in the closure cost estimate.

The new reclamation plan includes a pond. This pond is necessary because the mine is now too large to support 100% backfilling (as planned in the April 2019 permit). The pond will have 3:1 or 2:1 slopes around the rim, has an area of 2.2 million square feet, an average depth of 34 feet, and a volume of 1.5 million cubic yards (306.5 million gallons, 941 acre-feet).

Figure 20-3 shows a plan view map of the reclaimed mine.

Figure 19-3: Plan View Map of the Reclaimed Mine



19.1.2 Reclaiming Waste Dumps

As seen in Figure 20-3, the new mining plan will have 7.5 million cubic yards of WRD near the pit in three separate WRDs. These waste dumps will be reclaimed as per ACOE requirements. This will require a stable slope and revegetation with native species. Placement of waste will include slope breaks and swales to simulate natural ground. Topsoil beneath the WRDs will be stockpiled for use as the post-closure soil cover. In total, 2.9 million square feet of waste dump surface will require reclamation and revegetation.

19.1.3 Revegetation

Revegetation will occur over all impacted areas. As required by law, vegetative soil (topsoil or otherwise) will be placed atop disturbed ground and revegetated with native species.

19.1.4 Closure Cost Estimation

Concurrent reclamation will be performed during operations as recommended by the ACOE. Concurrent reclamation activities include the placement of 9.5 million cubic yards of backfill. This concurrent reclamation will reduce the total environmental impact of the mine and result in “bond release” during operations, allowing the operation to access secured funds based on acres of land taken to final reclaimed state. Final reclamation is expected to take 1 1/2 years to complete and will use the existing mine fleet.

19.2 Water Considerations

Goldrich maintains a State of Alaska water right, issued in 1985, allowing the withdrawal of up to 3,000 gallons (11,360 liters) of water per minute for placer mining, and 72,000 gallons (272,550 liters) per day for lode mining. The water can be withdrawn from any of the local streams specified in the permit for use from April through October. The water right is maintained by paying an annual $50 administrative fee and by demonstrating some beneficial use of the water at least once in any 5-year period.

The mine has a secure water source from upgradient springs. There are no competing claims to water use.

Because the mine exploits a 100% placer deposit, there is no significant risk to water quality apart from suspended sediments. The mine will have a robust sediment management program including preventive measures (erosion control best management practices) and sediment ponds.

19.3 Other Environmental and Social Considerations

The Chandalar mine is located in a remote area. No communities are in the vicinity of the project. Since operations began, there has been no social or community opposition.

20.0 Capital and Operating Costs

Capital and operating costs for the project were estimated using vendor quotes when possible, Ms. Lane's and Dr. Harvey's professional experience and data, and InfoMine USA, Inc. (InfoMine) resources, including *Mining Cost Service 2020* (InfoMine, 2020) and *Mine and Mill Cost Service 2018* (InfoMine, 2018).

Ms. Lane and Dr. Harvey used the *Mine and Mill Cost Service 2018* resource to estimate per ton rates for operations requirements. More detailed estimation was not appropriate for a Preliminary Economic Model level of effort because the project has not been designed to a level that would support more detailed cost estimating.

Ms. Lane and Dr. Harvey has assumed the project is constructed over a six-month period with production ramping up steadily over the summer gravel washing season. All of the mining and wash plant equipment are off the shelf items that are readily available. The selected wash plant has a 3- to 6-month lead time for delivery.

20.1 Capital Costs

The majority of the capital costs occur during the first quarter of year 1, i.e., during the quarter preceding the start of operations. Additional capital costs are scheduled as items are needed throughout the mine life. The following mining, processing, and G&A items were included in the capital cost estimate:

- Goldrich plans to lease mining production and support equipment. Lease prices were estimated by assuming a down payment equal to 25% of the purchase price and a lease term varying from 16 to 20 quarters, depending on the piece of equipment and when it is needed on the project, with a 5% interest rate.
- Mining facilities include a heavy equipment shop, dry, cap magazine and ANFO storage bin, and fuel station. Costs for the heavy equipment shop and fuel station at $75/square foot are included. For the dry and cap magazine and ANFO storage bin, shipping containers would be used, so no costs were included.
- Process equipment includes feed, wash plant, water supply, and gold room.
- The following utilities were included:
  - 200-kilowatt diesel genset: one for each wash plant and one for the admin facilities and camp
  - Sewage treatment plant: sized for the number of personnel on site
  - Potable water treatment
  - Fresh water pond
  - Fresh water pumps
  - Air compressor
- The following construction activities/items were included:
  - Installation labor
  - Concrete
  - Piping
  - Structural steel

- Instrumentation
- Insulation
- Electrical
- Coatings and sealants
- Spares and first fills
- Engineering/management

- G&A items include:
  - Survey: Goldrich already has survey equipment, so no costs were included
  - Guard house/security: this is already available on site, so no costs were included
  - Office: this is already available on site, so no costs were included
  - Warehouse: this is already available on site, so no costs were included
  - Fire Protection
  - Camp
  - Startup Training
  - Emergency vehicle/supplies
  - Winter road construction
  - Reclamation bond – uses the State of Alaska Department of Natural Resources State Wide Bond Pool, which includes $112.50/disturbed acre refundable deposit and $37.50/disturbed acre nonrefundable bond pool annual fee.
- Development included pioneering, clearing, grubbing, access road improvements, and haul road construction, assumed to be 5,000 feet of new or improved haul roads.
- Mine equipment was assumed to be salvageable at the end of the project. Recovery of 10% of the initial equipment price was included at the end of the project or when the equipment was no longer in use.

Working capital was estimated to be 2 months' operating costs. The working capital was estimated to be recovered the year after production ends. Sustaining capital was estimated as 10% of the mobile equipment cost per year. Capital contingency was set at 25%.

Owner's costs such as permitting, land, exploration, metallurgical testing, and feasibility studies were not included in the cash flow. Reasons for omitting these costs include:

- the mine has recently been operated and has existing permits, albeit ones that will need to be updated
- metallurgical testing is not needed due to production records and reconciliation to the resource model
- limited additional exploration/in-fill drilling is needed to convert inferred resources to measured or indicated; a plan for completing approximately 20 drill holes is under review by the company
- A pre-feasibility study will be completed prior to production.

The capital costs were scheduled by quarter but are summarized in Table 20-1 by year.

Table 20-1: Summary of Chandalar Mine Capital Costs (millions)

Capital Cost Item	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Total
Development	$2.78	$6.34	$4.63	$0.28					**$14.03**
Mine	$4.74	$1.97	$1.55	$1.55	$1.55	$1.30	$0.66		**$13.32**
Process	$1.57	$1.33							**$2.90**
G&A	$2.29	$0.03							**$2.32**
Freight, Taxes, and Insurance	$1.97								**$1.97**
Sustaining			$0.50						**$0.50**
Working	$2.27							($2.27)	**$0.00**
Contingency	$2.15	$0.83	$0.39	$0.39	$0.39	$0.33	$0.16		**$4.63**
Total	**$17.77**	**$10.49**	**$7.07**	**$2.22**	**$1.94**	**$1.63**	**$0.82**	**($2.27)**	**$39.67**

20.2 Operating Costs

Operating cash costs are based on a surface mine plan, haul cycle analysis, drill and blast cost analysis, with delivery to the remote mining site by either air (landing strip at the mine site) or by Cat train. Remote labor rates and burdens were used that are consistent with other remote mining operations in the arctic region of Alaska.

- Mining production equipment hours were estimated from the equipment productivity estimates (see Table 16-3), the scheduled process material and waste tonnage, and the number of pieces of equipment required.
- Mining support equipment hours were calculated from the number of pieces of equipment times the operating hours per day, assuming utilization of 90% and availability of 95%, times the operating days per year.
- Blasting materials requirements were determined in a drilling and blasting schedule that used the parameters and assumptions detailed in Table 20-2.

Table 20-2: Chandalar Mine Drilling and Blasting Parameters

Constants		
ANFO Density	0.028	tons/ft3
ANFO powder factor – mineralized material	0.45	lb/ton
ANFO powder factor - waste	0.36	lb/ton
bench height	20	ft
rock density	0.056	tons/ft3
drilling rate	2	feet/min
drill availability %	0.9	
minutes used/hr	50	
available hours per shift	9	
blasthole depth	20	ft
blasthole diameter	9.88	inch
rig type	rotary crawler	
rod length	45	ft

Constants		
ANFO thickness	16	ft
ANFO setup min/hole	15	min
Calculated		
blasthole diameter	0.82	ft
blasthole volume	10.65	ft3
ANFO volume	8.52	ft3
ANFO weight	0.24	tons
ANFO weight	478.61	lb
tons of rock blasted per hole	1055.15	tons/hole
vol of rock blasted per hole	695.37	bcy/hole
drillhole grid spacing	30.64	ft

- Operating hours for the plant were assumed to be 24 hours per day, 7 days per week, for 120 days per year.
- Reagent usage was estimated from equipment specifications and Dr. Harvery's experience and is summarized in Table 20-3.

Table 20-3: Summary of Chandalar Mine Estimated Reagent Usage

Item	Unit Consumption	Units
Water		
Diesel Water Pump	11.9	gal/hr
Consumables		
Maintenance Items	10%	equip cost
Fresh Water	66.0	m3/h
Food	20	$/day/person
Heating Oil	8,000	gal
Gasoline	6,500	gal
Misc Op Supplies	100	$/person

- Manpower for the mine and processing facility includes hourly-rate employees and salaried employees, who are generally superintendents and professional personnel. The number of required equipment operators was estimated using the quantities of equipment required, the quantity of personnel per piece of equipment, and the number of shifts per day. Numbers of required processing and salaried personnel were estimated based on Ms. Lane and Dr. Harvey 's experience and similar mines. A burden factor of 35% was added to all labor for fringe benefits, holidays, vacation and sick leave, insurances, etc. Some salaried personnel were assumed to work year-round, while others were assumed to work only during the mining season. Overtime of time and a half was applied to all hourly rate hours over 40 per week. A summary of the manpower requirements is provided in Table 20-4.

Table 20-4: Summary of Chandalar Mine Manpower Requirements

Labor	Quantity	Units
Mine		
Hourly		
Drillers	1	per machine
Blasters	1	per machine
Loader Operators	2	per machine
Truck Drivers	2	per machine
Excavator Operators	2	per machine
Scraper Operators	2	per machine
Grader Operators	1	per machine
Dozer Operators	2	per machine
Water Truck Operators	1	per machine
Mechanics	4/8	per day
Electricians	1	per day
Laborers/Maintenance	4/8	per day
Salaried		
Shift Foreman	2	ea
Engineer	1	ea
Geologist	1	ea
Technician	1	ea
Processing		
Hourly		
Plant Operator	3	per wash plant
Laborer	1	per wash plant
Salaried		
Maintenance Foreman	1	ea
Camp		
Hourly		
Cook	2	wash plant
Admin		
General Manager	1	ea
Environmental Specialist	1	ea
Purchasing Agent	1	ea
Accountant	1	ea
Security Guard	1	ea
Clerk	1	ea

- Administrative operating costs include services and supplies.

Annual and unit operating costs for equipment and facilities were obtained from recent quotes from suppliers and GRE internal database.

Operating contingency was set at 15%.

The operating costs summarized by year are shown in Table 20-5 and Table 20-6 (unit costs).

The cash operating costs per fine ounce of Au is $644, and the all-in sustaining cost per fine ounce of Au is $796.

Table 20-5: Summary of Chandalar Mine Operating Costs (millions)

Operating Cost Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Mine	$5.37	$8.18	$6.23	$9.56	$3.67	$0.00	$0.00	**$33.02**
Process	$1.17	$1.75	$1.75	$1.75	$4.05	$4.24	$1.55	**$16.28**
Camp	$0.26	$0.40	$0.33	$0.33	$0.33	$0.33	$0.14	**$2.10**
Transportation	$0.49	$0.50	$0.50	$0.50	$0.50	$0.50	$0.18	**$3.17**
Reclamation/Closure	$0.00	$0.00	$0.00	$0.00	$2.20	$3.79	$0.07	**$6.06**
Owner and G&A	$1.28	$1.36	$1.31	$1.28	$1.23	$1.10	$0.53	**$8.09**
Contingency	$1.29	$1.83	$1.52	$2.01	$1.80	$1.50	$0.37	**$10.31**
Total	**$9.86**	**$14.02**	**$11.64**	**$15.44**	**$13.78**	**$11.46**	**$2.84**	**$79.04**

Table 20-6: Summary of Chandalar Mine Unit Operating Costs

Operating Cost Item	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Mine	$/total mined bcy	$3.76	$1.75	$1.50	$3.42	$11.98			**$2.47**
Process	$/process bcy	$3.92	$1.76	$1.75	$1.82	$4.22	$4.42		**$2.97**
Camp	$/process bcy	$0.85	$0.40	$0.33	$0.34	$0.34	$0.34		**$0.38**
Transportation	$/process bcy	$1.65	$0.50	$0.50	$0.52	$0.52	$0.52		**$0.57**
Reclamation/Closure	$/process bcy	$0.00	$0.00	$0.00	$0.00	$2.29	$3.95		**$1.10**
Owner and G&A	$/process bcy	$4.27	$1.36	$1.31	$1.33	$1.28	$1.15		**$1.47**
Contingency	$/process bcy	$5.69	$2.79	$2.21	$2.13	$1.87	$1.56		**$1.87**

21.0 Economic Analysis

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under 17 CFR §229. This IA is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under 17 CFR §229. Readers are advised that there is no certainty that the results projected in this Preliminary Economic Assessment will be realized.

Ms. Lane used the 24-month trailing average for gold prices through May 31, 2021 of $1,650/oz for the base case analyses. The projected metal recovery rates are 100% with a ratio of fine gold to raw gold of 84%. The 100% recovery from the wash plants is based on the use of recoverable gold assays for the feed material; recovery losses are accounted for in the feed grade calculations as is typical of placer mining. Smelter charges of 3.6% were applied. Base case gold revenues are summarized in Table 21-1.

Table 21-1: Summary of Gold Revenues

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Raw Gold Processed (1000s)	11.09	47.83	33.33	27.19	16.53	7.58	2.71	**146.26**
Fine Gold Recovered (1000s)	9.32	40.17	28.00	22.84	13.89	6.36	2.28	**122.85**
Au Revenue (millions)	$15.37	$66.29	$46.20	$37.68	$22.91	$10.50	$3.76	**202,709.71**
Smelter Charges (millions)	($0.55)	($2.39)	($1.66)	($1.36)	($0.82)	($0.38)	($0.14)	**(7,297.55)**
Net Revenue (millions)	**$14.82**	**$63.90**	**$44.53**	**$36.33**	**$22.09**	**$10.12**	**$3.62**	**195,412.16**

Ms. Lane included depreciation and depletion deductions from the income before taxes to obtain taxable income. Depreciation was calculated using straight line depreciation over 5 years. Depletion allowance was calculated as 15% of revenues up to a maximum of 50% of before-tax income minus depreciation. Federal tax at 21% was applied to the taxable income, and Alaska mining license tax and production royalty were applied to the taxable income. The taxes were deducted from the taxable income, then the depreciation and depletion allowance were added back from taxable income to obtain net cash flows after taxes.

Note: Ms. Lane is not an expert in taxes and relied on information provided by Goldrich and obtained from on-line searches of U.S. and Alaska tax codes to generate a tax model for the project. The calculations are based on the tax regime as of the date of this 2021 IA. The tax calculations should be considered approximations because actual tax estimates involve complex calculations that can be accurately determined only during operations.

After-tax net present value (NPV) @5%, NPV@7%, NPV@9%, and internal rate of return (IRR) were calculated from the net after-tax cash flow.

Taxes, depreciation, and depletion were calculated on a yearly basis. Table 21-2 summarizes the economic model, and shows the after-tax NPV and IRR results.

The economic model indicates favorable after-tax NPV@5% of $63.7 million with an IRR of 139%, at a gold price of $1,650/oz.

The tax calculation should be considered conservative, without consideration for exploration costs incurred, corporate tax philosophy, and other factors.

Table 21-2: Summary of Economic Model

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mining Production									
High Grade Mineralized Material (1000s bcy)	454.60	994.68	999.53	965.31	176.94				**3,591.06**
High Grade Raw Au oz (1000s)	16.65	47.83	34.57	27.35	4.98				**131.37**
High Grade Au Grade (opy)	0.037	0.048	0.035	0.028	0.028				**0.037**
Low Grade Mineralized Material (1000s bcy)	130.29	355.31	458.37	897.71	45.12				**1,886.80**
Low Grade Raw Au oz (1000s)	1.03	2.76	3.57	7.09	0.45				**14.89**
Low Grade Au Grade (opy)	0.008	0.008	0.008	0.008	0.010				**0.008**
Waste BCY during Mining (1000s bcy)	843.00	3,322.57	2,697.79	933.29	84.56				**7,881.22**
Pre-Strip Waste from Existing Lower Dump (1000s bcy)	660.00	376.85	0.00	0.00	0.00				**1,036.85**
Stripping Ratio	2.6	2.7	1.9	0.5	0.4				**1.6**
Total Mined Material (1000s bcy)	2,087.89	5,049.42	4,155.69	2,796.31	306.62				**14,395.93**
Process Production									
High Grade Mineralized Material Processed (1000s bcy)	299.40	994.68	960.00	960.00	176.94				**3,391.02**
High Grade Raw Au Oz Processed	11.09	47.83	33.33	27.19	4.98				**124.42**
Process from Stockpile (1000s bcy)					783.06	960.00	343.77		**2,086.83**
Process from Stockpile Raw Au oz					11.55	7.58	2.71		**21.84**
Revenue									
Fine Au Oz (1000s) (84% of Raw Au Oz)	9.32	40.17	28.00	22.84	13.89	6.36	2.28		**$123**
Au Revenue (1000s) (@ $1,580/fine Au Oz)	$15,372	$66,287	$46,195	$37,684	$22,912	$10,499	$3,760		**$202,710**
Smelter Charges (1000s)	$553	$2,386	$1,663	$1,357	$825	$378	$135		**$7,298**
Net Revenue (1000s)	**$14,819**	**$63,900**	**$44,532**	**$36,327**	**$22,088**	**$10,121**	**$3,624**		**$195,412**
Operating Costs (1000s)									
Mine	$5,373	$8,179	$6,230	$9,562	$3,673	$0	$0		**$33,017**
Process	$1,174	$1,753	$1,753	$1,753	$4,054	$4,242	$1,552		**$16,282**
Reclamation/Closure					$2,197	$3,792	$69		**$6,058**
Transportation	$494	$500	$500	$500	$500	$500	$179		**$3,173**
G&A	$1,280	$1,357	$1,308	$1,282	$1,230	$1,105	$530		**$8,092**
Camp	$255	$398	$328	$328	$328	$328	$137		**$2,105**
Operating Cost Contingency	$1,286	$1,828	$1,518	$2,014	$1,797	$1,495	$370		**$10,309**
Total Operating Costs	**$9,863**	**$14,016**	**$11,637**	**$15,439**	**$13,781**	**$11,463**	**$2,837**		**$79,036**
Net Income (Loss) before DD&A (1000s)	**$4,956**	**$49,884**	**$32,896**	**$20,888**	**$8,307**	**($1,341)**	**$787**		**116,377**
Tax Deductions									
Mine Development Amortization	($1,002)	($6,917)	($12,609)	($26,229)	($7,351)	$0	$0	$0	**($54,107)**
Depreciation	($2,420)	($1,754)	($1,266)	($878)	($489)	$0	$0	$0	**($6,807)**

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Alaska State Tax	$0	$0	$0	($1,156)	$0	($26)	$0	($57)	**($1,239)**
Total Tax Deductions	**($3,422)**	**($8,672)**	**($13,875)**	**($28,262)**	**($7,840)**	**($26)**	**$0**	**($57)**	**($62,153)**
Taxable Income (Loss) before Depletion	**$1,534**	**$41,213**	**$19,021**	**($6,218)**	**$467**	**($1,341)**	**$787**	**$0**	**$55,462**
Depletion Allowed	($767)	($9,393)	($6,546)	$0	($233)	$0	($394)	$0	**($17,334)**
Taxable Income (Loss) After Depletion	**$767**	**$31,819**	**$12,474**	**($6,218)**	**$233**	**($1,341)**	**$394**	**$0**	**$38,128**
Loss Carry Forward	($453)	($29,507)	($9,040)	$0	$0	$0	$0	$0	**($39,000)**
Federal Taxes	$0	$0	$1,273	$0	$49	$0	$17	$0	**$1,338**
Alaska Taxes	$0	$0	$1,156	$0	$26	$0	$57	$0	**$1,239**
Total Taxes	**$0**	**$0**	**$2,429**	**$0**	**$75**	**$0**	**$73**	**$0**	**$2,577**
After Tax Operating Cash Flow	**$4,956**	**$49,884**	**$30,467**	**$20,888**	**$8,232**	**($1,341)**	**$714**	**$0**	**$113,800**
Capital Costs									
Development	$2,784	$6,3367	$4,629	$283					**$14,033**
Mine	$4,737	$1,966	$1,553	$1,553	$1,553	$1,302	$656		**$13,320**
Plant	$1,571	$1,328							**$2,899**
G&A	$2,293	$26							**$2,320**
Freight, Taxes, and Insurance	$1,965								**$1,965**
Sustaining			$500						**$500**
Working	$2,266					$0	$0	($2,266)	**$0**
Contingency	$2,150	$830	$388	$388	$388	$326	$164		**$4,635**
Total Capital Costs	**$17,767**	**$10,487**	**$7,070**	**$2,224**	**$1,941**	**$1,628**	**$820**	**($2,266)**	**$39,671**
Net Cash Flow	**($12,811)**	**$39,398**	**$23,396**	**$18,664**	**$6,291**	**($2,969)**	**($106)**	**$2,266**	**$74,129**

Table 21-3: Summary of Economic Results

After-tax NPV@5% (millions)	After-tax NPV@7% (millions)	After-tax NPV@9% (millions)	After-tax Cash Flow (millions)	After-tax IRR	Payback Years
$63.7	$60.7	$57.9	$72.0	139%	1.33

Key economic results with a summarized gold price sensitivity analysis are shown in Table 21-4.

Table 21-4: Key Economic Results

Parameter	Base Case $1,650 Gold	Gold Price Sensitivity Analysis		
		$1,500	$2,000	$2,500
State Royalties:	3%			
Undiscounted Pre-Tax Net Cash Flow:	$75 million	$57 million	$116 million	$175 million
Pre-tax NPV@5%:	$67 million	$51 million	$104 million	$158 million
Pre-tax NPV@7%:	$64 million	$49 million	$100 million	$151 million
Pre-tax NPV@9%:	$61 million	$47 million	$963 million	$146 million
Pre-tax IRR:	256.%	187.8%	482.7%	1243.4%
After-tax NPV@5%:	$64 million	$50 million	$92 million	$129 million
After-tax NPV@7%:	$61 million	$48 million	$88 million	$123 million
After-tax NPV@9%:	$58 million	$45 million	$84 million	$118 million
After-tax IRR:	138.8%	112.4%	195.5%	275.0%
Undiscounted After-tax Net Cash Flow:	$72 million	$57 million	$103 million	$145 million
After-tax Payback Period:	1.33	1.44	1.19	1.08
All-in Sustaining Costs:	$799/Au ounce			
All-in Costs:	$1064/Au ounce			
Total Operating Costs:	$646/Au ounce			

21.1 Sensitivity Analyses

Ms. Lane evaluated the after-tax NPV@7% and IRR sensitivity to changes in gold price, capital costs, and operating costs. The results are shown in Table 21-5, Figure 21-1, Table 21-6, and Figure 21-2, respectively. A more detailed sensitivity analysis to gold price, showing extreme gold prices is also provided in Figure 21-3.

Table 21-5: NPV@7% Sensitivity to Gold Price, Capital Costs, and Operating Costs

Variable	NPV@7%				
	80%	90%	100%	110%	120%
Capital Cost	$65.26	$62.99	$60.72	$58.44	$56.17
Operating Cost	$72.05	$66.67	$60.72	$54.50	$46.80
Gold Price	$29.36	$46.14	$60.72	$73.64	$86.08

Figure 21-1: NPV@7% Sensitivity to Gold Price, Capital Costs, and Operating Costs

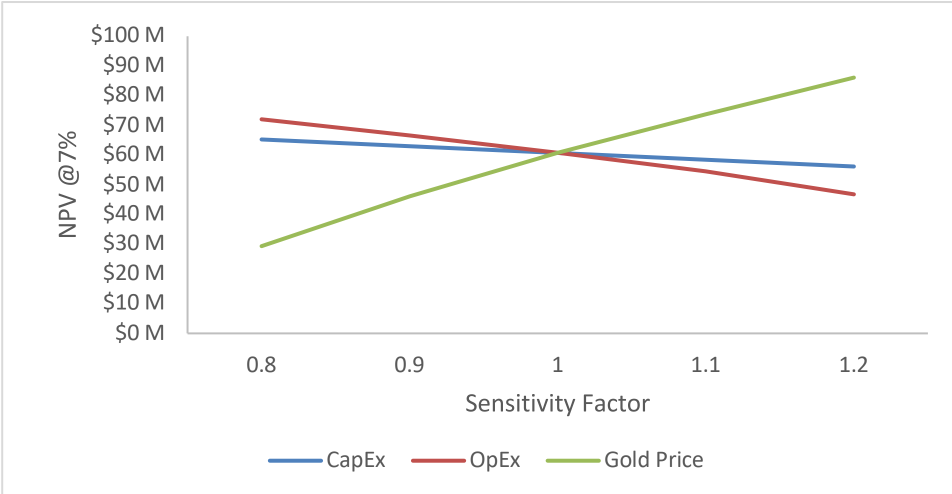


Table 21-6: IRR Sensitivity to Gold Price, Capital Costs, and Operating Costs

Variable	IRR				
	80%	90%	100%	110%	120%
Capital Cost	173.3%	154.5%	138.8%	125.4%	113.8%
Operating Cost	171.3%	154.5%	138.8%	124.1%	108.4%
Gold Price	77.1%	109.5%	138.8%	167.2%	192.4%

Figure 21-2: IRR Sensitivity to Gold Price, Capital Costs, and Operating Costs

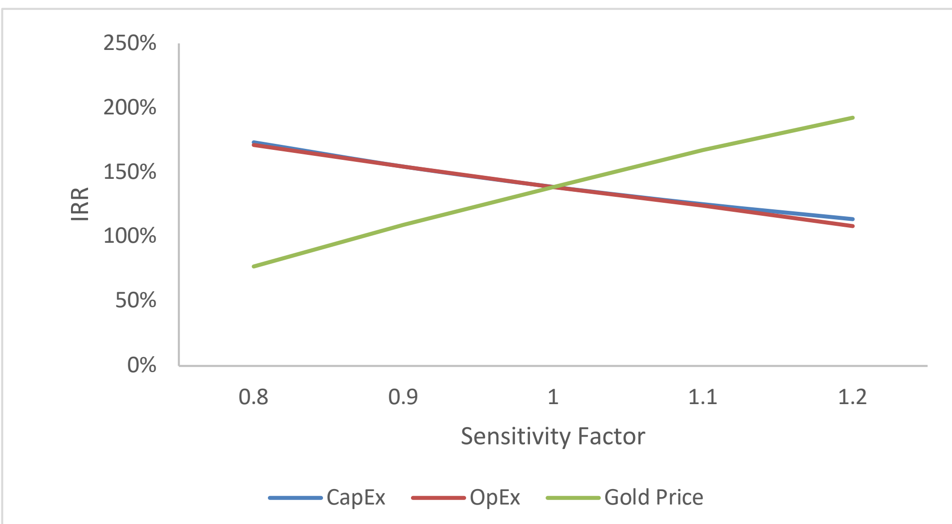
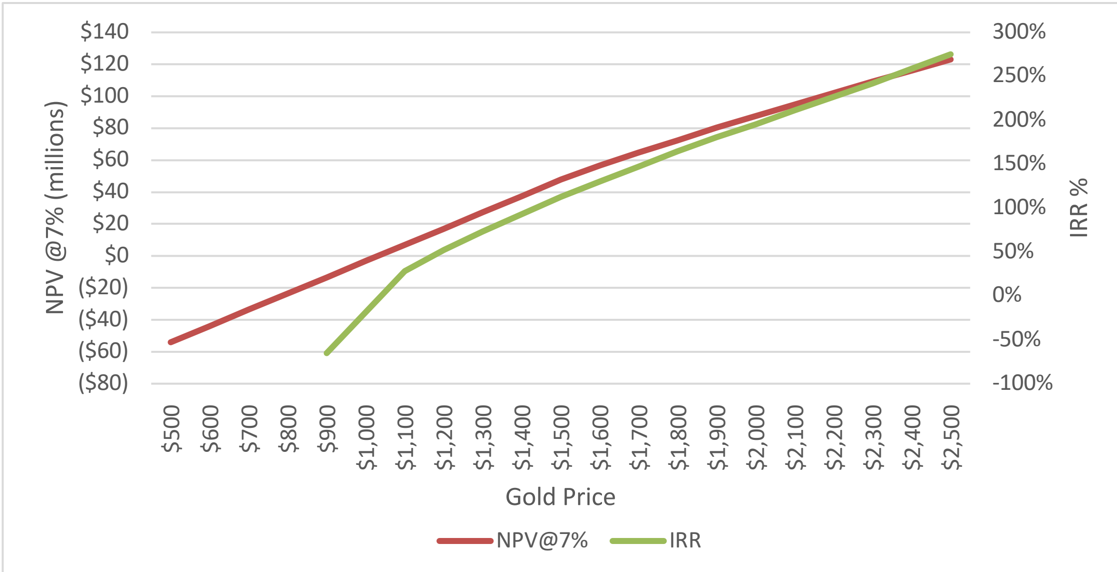


Figure 21-3: NPV@7% and IRR Sensitivity to Gold Price



The results indicate after-tax NPV@7% and IRR are extremely sensitive to changes gold price and are moderately sensitive to changes in capital and operating costs. A positive valuation, however, is maintained across a wide range of sensitivities on key assumptions.

21.2 CONCLUSIONS OF ECONOMIC MODEL

The project economics shown in this IA are favorable, providing positive NPV at varying gold prices, capital costs, and operating costs.

22.0 Adjacent Properties

There are six other placer deposits in the Chandalar district. Combined, they have produced an estimated 104,000 ounces of gold from the early 1900s to the present day, with approximately 73,000 ounces coming from Little Squaw. Other significant placers include Big Creek, St. Mary's Creek, and Tobin Creek. The district has also produced an estimated 8,200 ounces of gold from four lode deposits: Mikado, Crystal, Little Squaw, and Summit (Walters, 2020).

23.0 Other Relevant Data and Information

Section 27, References, provides a list of documents that were consulted in support of this report. No further data or information is necessary, in the opinion of the authors, to make the Report understandable and not misleading.

24.0 Interpretation and Conclusions

24.1 Interpretation

The Chandalar district is located in a remote portion of Alaska in the Brooks Range. The district has hosted both placer and gold production since the early 1900s. The remote location and harsh winters have limited access, making large scale exploration and mine production difficult. This moderated the success of past efforts. The more recent access to the site via winter road and modern air strips have opened up the area to modern exploration and mining production methods. The Chandalar project hosts both placer and lode gold mineralization; the full potential of each have not been fully tested.

24.2 Conclusions

Capital costs for the project include mining production and support equipment leases that assume 25% down payment of the purchase price and a lease term varying from 20 to 26 quarters, depending on the piece of equipment and when it is needed on the project, at 5% interest rate; mining facilities; and site development. The estimated capital costs total $25.6 million, with initial capital of $15.1 million.

Operating cash costs are based on a surface mine plan, haul cycle analysis, drill and blast cost analysis, with delivery to the remote mining site by either air (landing strip at the mine site) or by Cat train. Remote labor rates and burdens were used that are consistent with other remote mining operations in the arctic region of Alaska. The estimated operating costs total $95.2 million, with a cash operating cost of $646/fine Au oz and an all-in sustaining cost of $799/fine Au oz.

Ms. Lane prepared an economic evaluation of the project using a gold price of $1,650/Au oz and assuming a raw gold recovery of 100% and ratio of find gold to raw gold of 84%. The 100% recovery from the wash plants is based on the use of recoverable gold assays for the feed material; recovery losses are accounted for in the feed grade calculations as is typical of placer mining. Smelter charges of 3.6% were applied. Ms. Lane included an order of magnitude, generalized tax calculation that included straight-line depreciation, depletion allowance, federal tax at 21% applied to the taxable income, and Alaska corporate, mining license tax, and production royalty applied to the taxable income.

Sensitivity analyses indicate extreme sensitivity to gold price and moderate sensitivity to both capital and operating costs. A positive valuation, however, was maintained across a wide range of sensitivities on key assumptions. Key economic results with a summarized gold price sensitivity analysis are shown in Table 24-1.

Table 24-1: Key Economic Results

Parameter	Base Case $1,650 Gold	Gold Price Sensitivity Analysis $1,500	$2,000	$2,500
State Royalties:	3%			
Undiscounted Pre-Tax Net Cash Flow:	$75 million	$57 million	$116 million	$175 million
Pre-tax NPV@5%:	$67 million	$51 million	$104 million	$158 million
Pre-tax NPV@7%:	$64 million	$49 million	$100 million	$151 million
Pre-tax NPV@9%:	$61 million	$47 million	$963 million	$146 million
Pre-tax IRR:	256.%	187.8%	482.7%	1243.4%
After-tax NPV@5%:	$64 million	$50 million	$92 million	$129 million

Parameter	Base Case $1,650 Gold	Gold Price Sensitivity Analysis		
		$1,500	$2,000	$2,500
After-tax NPV@7%:	$61 million	$48 million	$88 million	$123 million
After-tax NPV@9%:	$58 million	$45 million	$84 million	$118 million
After-tax IRR:	138.8%	112.4%	195.5%	275.0%
Undiscounted After-tax Net Cash Flow:	$72 million	$57 million	$103 million	$145 million
After-tax Payback Period:	1.33	1.44	1.19	1.08
All-in Sustaining Costs:	$799/Au ounce			
All-in Costs:	$1064/Au ounce			
Total Operating Costs:	$646/Au ounce			

The project economics shown in this IA are favorable, providing positive NPV at varying gold prices, capital costs, and operating costs.

24.3 Risks

This IA is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under 17 CFR §229.1300 – (Item 1300). Readers are advised that there is no certainty that the results projected in this initial assessment will be realized. There is inherent engineering and metallurgical risk in all projects at this early stage of development, which are reduced as metallurgical test work and engineering studies, including the prefeasibility and feasibility studies, progress. This project, as with others, would be negatively impacted should gold recovery prove to be less than that used in the study (due to clay or other factors) or positively impacted if higher grades are encountered. Similarly, if further geotechnical or reclamation requirements change, the project could be impacted positively or negatively. The largest likely impact would be future changes in the price of gold.

The site has low environmental and permitting risk; however, prior reclamation plans had nearly-complete backfilling. There is a small probability that similar backfilling would be required. This would result in higher closure costs.

The final topographic surface Ms. Lane used for the resource estimate stitched together the final 2018 survey completed by GNP on 21 September 2018, which is limited to the areas immediately surrounding the mined areas, and the translated 2019 FODAR survey. This stitched together surface constitutes a small risk to the estimated resources, but because the 2018 surface would be lower than the 2019 surface due to backfilling that may have occurred in 2019 and due to the fact that there was no mining conducted in 2019, Ms. Lane believes the estimated remaining in-situ resource would not change with the 2019 topographic surface but that there could be additional waste backfilled above the estimated resource.

Similarly, the stitched together mining extents of mining could represent a risk if the surfaces provided to GRE do not fully encompass the mining completed by GNP.

25.0 Recommendations

The Chandalar placer deposit has been successfully mined, and approximately 146,000 raw ounces of gold remain and appear to be recoverable with conventional mining and placer gold wash plant operation. In order to begin operations, permits need to be acquired, either through negotiation with NyacAU, or new operating permits must be obtained.

A detailed mining plan should be created to carefully plan material movement to concurrently reclaim the mine with operations.

Existing permits should be maintained.

A geologic model of lithology and clays would be helpful to optimize the mine plan.

Ms. Lane recommends the company proceed to a Pre-Feasibility Study (PFS) concurrent with obtaining new operating permits. Once the operating permits are in place and the PFS has been published, Ms. Lane recommends the company proceed with detailed costing, planning, and scheduling need to restart operations.

The mine will require modified permits with the ACOE. These should commence as soon as possible to avoid delays.

Ms. Lane recommends compiling all existing geologic data, property-wide, and conducting new mapping and geophysical surveys of high priority targets in preparation for drill campaigns.

The coordinate system for the project is a local Imperial coordinate system; some surveying has been conducted in this local coordinate system while others have been performed in UTM NAD83. Ms. Lane recommends adding the UTM coordinates to the drill hole database and using that grid and metric measurements for all project work going forward. This will reduce the possibility of conversion errors and facilitate reporting to international standards.

26.0 References

Ashworth, K. K. 1983. *Genesis of Gold Deposits at the Little Squaw Mines, Chandalar Mining District, Alaska.* 1983.

Ashworth, Kate. 1984. *Fluid Inclusion Study of the Eneveloe Vein, Chandalar Mining District: Private Report, Chandalar Development Associates.* 1984.

—. 1983. *Genesis of Gold Deposits and the Little Squaw Mines, Chandalar Mining District, Alaska: Unpublished Masters of Science Thesis, Western Washington University, Bellingham.* 1983.

Barker, J. C., Keener, J. O. and Murray, R. B. 2009. *Evaluation of the Chandalar Mining Property, with a Preliminary Assessment by Paul L. Martin.* 2009.

Barker, James C. and Bundtzen, Thomas K. 2004. *Gold Deposits of the Chandalar Mining District, Northern Alaska: An Information Review and Recommendations (May 1).* 2004.

Barker, James C., Keener, Jeffrey O. and Murray, Robert B. 2009. *Placer Exploration in the Chandalar Mining District: a case for bulk mineable gold placer.* 2009. presentation to AMA.

Bolin, D. S. 1984. *Geology and Lode Gold Deposits of the Chandalar Mining District, Brooks Range, Alaska (Golder Associates unpublished report).* 1984.

Brosgé, W. P. and Reiser, H. N. 1970. *Geochemical Reconnaissance in the Wisemand and Chandalar Mining Districts and Adjacent Region, Southern Brooks Range, Alaska: U.S. Geological Survey Professional Paper 709.* 1970.

—. 1964. *Geologic Map and Section of the Chandalar Quadrangle, Alaska: U.S. Geological Survey Miscellaneous Map I-375.* 1964.

Bundtzen, T. K. and Laird, G. M. 2007. *Geology of the Chandalar Mining District, Central Brooks Range, Northern Alaska, Report to Little Squaw Gold MiningCo.* 2007.

Chipp, E. R. 1970. *Geology and Geochemistry of the Chandalar Area, Brooks Range, Alaska: Alaska Division of Mines and Minerals Geologic Report 42.* 1970.

CIM. 2010. *Definition Standards for Mineral Resources and Mineral Reserves.* s.l. : CIM Standing Committe on Reserve Definitions, 2010.

—. 2014. *Definition Standards for Mineral Resources and Mineral Reserves.* s.l. : CIM Standing Committe on Reserve Definitions, 2014.

Dillon, J. T. 1982. *Source of Lode and Placer Gold Deposits of he Chandalar and Upper Koyukuk District, Alaska: Alaska Division of Geological an dGeophysical Surveys Open Fiel Report 158.* 1982.

—. 1989. Structure and Stratigraphy of the Southern Brooks Range and Northern Koyukuk Basin Near the Dalton Highway. [ed.] C. G. Mull and K. E. Adams. *Dalton Highway, Yukon River to Prudhoe Bay, Alaska: Alaska Divison of Geological and Geophysical Surveys Guidebook 7.* 1989, Vol. 2, pp. 157-187.

Dillon, J. T., Lamal, K. K. and Huber, J. A. 1989. Gold Deposits in the Upper Koyukuk and Chandalar Mining Districts. [ed.] C. G. Mull and K. E. Adams. *Dalton Highway, Yukon River to Prudhoe Bay, Alaska; Alaska Division of Geological and Geophysical Surveys Guidebook 7.* 1989, Vol. 2, pp. 195-201.

Dillon, J. T., Reifenstuhl, R. R. and Harris, G. W. 1996. *Geologic Map of the Chandalar C-5 Quadrangle, Southeastern Brooks Range, Alaska: Alaska Division of Geological and Geophysical Surveys Professional Report 104.* 1996.

Duke, N. 1975. *Structural Interpretation of the Chandalar Mining District, Alaska; Unpublished Callahan Mining Company Data.* 1975.

Dusel-Bacon, C. 1994. Metamorphic History of Alaska. [ed.] G. Plafker and H. C. Berg. *The Geology of Alaska: Boulder, Colorado, Geological Society of America, The Geology of North America.* 1994, Vols. G-1, pp. 495-533.

EPA. 2012. *A Function-Based Framework for Stream Assessment and Restoration Projects.* s.l. : US Environmental Protection Agency, 2012. EPA 843-K-12-006.

Fitch, Gary. 1997. *Placer Gold Drilling and Avaluation, Little Squaw and Big Squaw Creeks, Chandalar Mining District.* 1997.

Fritts, C. E. 1970. *Geology and Geochemistry of the Cosmos Hills, Ambler River and Shungnak Quadrangles, Alaska: Alaska Division of Mines and Geology Geologic Report 39.* 1970.

Fritts, C. E., Eakins, G. R. and Garland, R. E. 1971. *Geology and Geochemistry near Walker Lake, Southern Survey Pass Quadrangle, Arctic Alaska: Alaska Division of Geological dn geophysical Surveys Annual Report.* 1971. pp. 19-26.

Gillerman, V. S. 2012. *Final Petrographic Report: Chandalar Project, Alaska.* 2012.

Goldfarb, R. J., et al. 2005. Distribution, Character, and Genesis of Gold Deposits in Metamorphic Terranes. *Economic Geology.* 2005, Vol. 100th Anniversary Volume, pp. 407-450.

Goldfarb, Y. I. 2007. Dynamic Classification of Alluvial Gold Placers in the Northeast of Russia. *Geology of Ore Deposits.* 2007, Vol. 49, No. 4, pp. 241-270. Original Russian Text © Yu.I. Goldfarb, 2007, published in Geologiya Rudnykh Mestorozhdenii, 2007, Vol. 49, No. 4, pp. 275–305.

Hitzman, M. W., Smith, T. E. and Proffett, J. M. 1982. *Bedrock Geology of the Ambler District, Southwestern Bokks Range, Alaska: Alaska Division of Geological and Geophysical Surveys Geologic Report 75.* 1982.

Hulstulid, W., Kuchta, M. and Martin, R. 2006. *Open Pit Mine Planning & Design.* Leiden : CRC Press, 2006.

InfoMine. 2018. *Mine and Mill Equipment Costs, An Estimator's Guide.* 2018.

—. 2020. *Mining Cost Service.* 2020.

Kuehner, S. 2013. *Microprobe Observations on Six Core Samples from Chandalar Mining District, Alaska.* 2013.

Lamal, K. K. 1984. *Fluid Inclusion Study Eneveloe Mine, Chandalar Mining District.* 1984.

Little Squaw Mining Co. 2006. *SEC.gov.* [Online] 2006. https://www.sec.gov/Archives/edgar/data/59860/000105291806000712/nr1506.htm.

Maddren, A. G. 1910. The Koyukuk-Chandalar Gold Region, Alaska. *USGS Bull. 442-G.* 1910, pp. 284-315.

Martin, P., et al. 2010. *Goldrich Mining Company Preliminary Assessment, Data Analysis and Mining Methods Summary, Little Squaw Creek Alluvial Gold Deposit, Chandalar, Alaska.* 2010.

Mendham, G. and Lam, C. 2018. *Chandalar Little Squaw Creek Placer Gold Project, Unpublished Report for Goldrich Mining Company.* 2018.

Mertie, Jr., J. B. 1925. Geology and Gold Placers of the Chandalar Mining District. *U.S. Geological Survey Bulleting 773E.* 1925, pp. 215-267.

Moore, T. E., et al. 1994. Geology of Northern Alaska. [ed.] G. Plafker and H. C. Berg. *The Geology of Alaska: Boulder, Colorado, Geological Society of America, The Geology of North America.* 1994, Vols. G-1, pp. 49-140.

Mull, C. G. 1989. Chapter 6 - Summary of Structural Style and History of the Brooks Range Deformation. [ed.] C. G. Mull and K. E. Adams. *Dalton Highway, Yukon River to Prudhoe Bay, Alaska: Alaska Division of Geological and Geophysical Surveys Guidebook 7.* 1989, Vol. 1, pp. 47-56.

Pacific Rim Geological Consulting, Inc. 2007. *Follow-up 2007 Geologic Mapping, Structural Analysis and Evaluation of Gold Deoosits in Chandalar Mining District, Northern Alaska, Confidential Report for Little Squaw Gold Mining Company.* 2007.

Pitard, F. F. 1993. *Pierre Gy's Sampling Theory and Sampling Practice. Heterogeneity, Sampling Correctness, and Statistical Process Control.* 1993.

Rasmussen, M. 2013. *Summary of Current Hardrock Exploration Views at Chandalar Mining District, Alaska.* 2013.

Reger, R. D. and Bundtzen, T. K. 1990. Multiple Glaciation and Gold-Placer Formation, Valdez Creek Valley, Western Clearwater Mountains, Alaska. *Alaska Division of Geological and Geophysical Surveys Professional Report 107.* 1990.

Rose, S.C., Pickthorn, W. J. and Goldfarb, R. J. 1988. Gold Mineralization by Metamorphic Fluids in the Chandalar Mining District, Southern Brooks Range - Fluid Inclusion and Oxygen Isotopic Evidence. [ed.] J. P. Galloway and T. D. Hamilton. *Geologic Studies in Alaska by the U.S. Geological Survey during 1987: U.S. Geological Survey Circular 1016.* 1988, pp. 81-84.

SME. 2011. *Mining Engineering Handbook.* [ed.] Peter Darling. s.l. : Society for Mining, Metallurgy, and Exploration, 2011.

Strandberg, Jr., Edward O. 1990. *Description of Properties, Chandalar mining District (January 27).* 1990.

Szumigala, D. J. and Hughes, R. A. 2005. *Alaska's Mineral Industry, 2004: Alaska Division of Geological & Geophysical Survey Special Report 59.* 2005.

Till, A. B. 1992. Detrital High Pressure/Low Temperature Metamorphic Mineral Assemblages in Early Cretaceous Sediments of the Foreland Basin of the Brooks Range, Alaska, and Implications for Orogenic Evolution. *Tectonophysics.* 1992, Vol. 11, pp. 1207-1223.

Turner, D. L., Forbes, R. B. and Dillon, J. T. 1979. K-Ar Geochronology of the Southern Brooks Range, Alaska. *Canadian Journal of Earth Sciences.* 1979, Vol. 16, No. 9, pp. 1989-1804.

USGS. 1956. *Aeromagnetic Map, Chandalar Mining District, Alaska.* 1956.

—. 1986. Mineral Deposit Models. [ed.] D. P. Cox and D. A. Singer. *United States Geological Survey Bulletin 1693.* 1986.

Vanderberg, W. O. 1936. Placer Mining in Nevada. *Nevada Bureau of Mines and Geology, Bulletin 27.* 1936.

Walters, D. 2020. *Chandalar District Gold Production 20200225, unpublished spreadsheet compiling historical gold production at Chandalar.* 2020.

Wiltse, M. W. 1975. *Geology of the Arctic Camp Deposit: Alaska Division of Geological and Geophysical Surveys Open File Report.* 1975.

Wolff, Ernest N. 1997. *Frank Yasuda and the Chandalar.* 1997.

CERTIFICATE OF QUALIFIED PERSON

I, Hamid Samari, PhD, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Revised and Amended Preliminary Economic Assessment, Chandalar Mine, Chandalar Gold District" with an effective date of May 31, 2021 and an Issue date of November 10, 2022 (the "IA"), DO HEREBY CERTIFY THAT:

1. I am a graduate of Azad University, Sciences and Research Branch, Tehran and received a PhD in Geology-Tectonics in 2000. I am also a graduate of Beheshti University, Tehran and received a MS in Geology-Tectonics in 1995 and a MS and a BS in Geology in 1991.
2. I am a Qualified Professional in the United States from the Mining and Metallurgical Society of America (MMSA) with special expertise in Geology with membership number MMSA 01519QP.
3. I have practiced in the area of geology, mining, and civil industry for over 23 years. I have worked for Azad University, Mahallat branch, as an assistant professor and head of the geology department for 19 years, for Tamavan consulting engineers as senior geologist for 12 years, and for Global Resource Engineering for about four years.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of 17 CFR §229.1300 – (Item 1300).
5. I visited the project on September 4 and 5, 2019.
6. I am responsible for Sections 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 7, 8, 9, 10, 11, and 12 of the IA.
7. I am independent of Goldrich Mining as described in 17 CFR §229.1300 – (Item 1300).
8. I have not previously worked on the Chandalar Mine.
9. I have read 17 CFR §229.1300 – (Item 1300). The Resource Estimate has been prepared in compliance with the 17 CFR §229.1300 – (Item 1300).
10. As of the effective date of the IA, to the best of my knowledge, information and belief, the IA contains all scientific and technical information that is required to be disclosed to make the IA not misleading.

Hamid Samari, PhD

"Hamid Samari"

Director of Geology

Global Resource Engineering, Ltd.

Denver, Colorado

Date of Signing: November 10, 2022

CERTIFICATE OF QUALIFIED PERSON

I, Jeffrey Todd Harvey, PhD, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Revised and Amended Preliminary Economic Assessment, Chandalar Mine, Chandalar Gold District" with an effective date of May 31, 2021 and an Issue date of November 10, 2022 (the "IA"), DO HEREBY CERTIFY THAT:

1. I am a Society of Mining Engineers (SME) Registered Member Qualified Professional in Mining/Metallurgy/Mineral Processing, #04144120.
2. I hold a degree of Doctor of Philosophy (PhD) (1994) in Mining and Mineral Process Engineering from Queen's University at Kingston. As well as an MSc (1990) and BSc (1988) in Mining and Mineral Process Engineering from Queen's University at Kingston.
3. I have practiced my profession since 1988 in capacities from metallurgical engineer to senior management positions for production, engineering, mill design and construction, research and development, and mining companies. My relevant experience for the purpose of this IA is as the test work reviewer, process designer, process cost estimator, and economic modeler with 25 or more years of experience in each area.
4. I have taken classes in mineral processing, heap leach design, cost estimation and mineral economics in university, and have taken several short courses in process development subsequently.
5. I have worked in mineral processing, managed production and worked in process optimization, and I have been involved in or conducted the test work analysis and flowsheet design for many projects at locations in North America, South America, Africa, Australia, India, Russia and Europe for a wide variety of minerals and processes.
6. I have supervised and analyzed test work, developed flowsheets and estimated costs for many projects including International Gold Resources Bibiani Mine, Ashanti Goldfields Obuasi Mine, Equinox Gold Castle Mountain Mine, Cluff Resources Agnes Mine, and others, and have overseen the design and cost estimation of many other similar projects.
7. I have worked or overseen the development or optimization of mineral processing flowsheets for close to one hundred projects and operating mines, including gold heap leach and stirred tank gold leaching processes.
8. I have been involved in or managed many studies including scoping studies, prefeasibility studies, and feasibility studies.
9. I have been involved with the mine development, construction, startup, and operation of several mines.
10. I have read the definition of "Qualified Person" set out in 17 CFR §229.1300 – (Item 1300) and certify that by reason of my education, affiliation with a professional organization (as defined in 17 CFR §229.1300 – (Item 1300)) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of 17 CFR §229.1300 – (Item 1300).
11. I have not visited the project.
12. I am responsible for Sections 1.10, 1.13, 13, and 17 of the IA.
13. I am independent of Goldrich Mining as described in 17 CFR §229.1300 – (Item 1300).
14. I have not previously worked on the Chandalar Mine.
15. I have read 17 CFR §229.1300 – (Item 1300). The Resource Estimate has been prepared in compliance with the 17 CFR §229.1300 – (Item 1300).

16. As of the effective date of the IA, to the best of my knowledge, information and belief, the IA contains all scientific and technical information that is required to be disclosed to make the IA not misleading.

Jeffrey Todd Harvey, PhD

"J. Todd Harvey"

President and Director of Process Engineering
Global Resource Engineering, Ltd.
Denver, Colorado
Date of Signing: November 10, 2022

CERTIFICATE OF QUALIFIED PERSON

I, Terre A Lane, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Revised and Amended Preliminary Economic Assessment, Chandalar Mine, Chandalar Gold District" with an effective date of May 31, 2021 and an Issue date of November 10, 2022 (the "IA"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Ore Reserves and Mining, #01407QP and a Registered member of SME - 4053005.
2. I hold a degree of Bachelor of Science (1982) in Mining Engineering from Michigan Technological University.
3. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. My relevant experience for the purpose of this IA is project management, mineral resource estimation, mine capital and operating costs estimation, and economic analysis with 25 or more years of experience in each area. My experience includes several placer deposits including the Livengood permafrost gold placer near Livengood Alaska.
4. I have created or overseen the resource estimation, mine design, capital and operating cost estimation, and economic analysis of well over a hundred open pit projects.
5. I have been involved in or managed several hundred studies including scoping studies, prefeasibility studies, and feasibility studies.
6. I have been involved with the mine development, construction, startup, and operation of several mines, including the Livengood placer deposit, a permafost placer 70 miles north of Fairbanks.
7. I have read the definition of "Qualified Person" set out in 17 CFR §229.1300 – (Item 1300) and certify that by reason of my education, affiliation with a professional organization (as defined in 17 CFR §229.1300 – (Item 1300)) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of 17 CFR §229.1300 – (Item 1300).
8. I visited the property on September 4 and 5, 2019.
9. I am responsible for Sections 1.1 - 1.3, 1.11, 1.12 1.14, 1.15, 1.18, 1.19, 1.20, 2, 3, 4.1 - 4.4, 5, 6, 14, 15, 16, 18, 19, and 21 – 27 of the IA.
10. I am independent of Goldrich Mining as described in 17 CFR §229.1300 – (Item 1300).
11. I have read 17 CFR §229.1300 – (Item 1300). The IA has been prepared in compliance with 17 CFR §229.1300 – (Item 1300).
12. As of the effective date of the IA, to the best of my knowledge, information and belief, the IA contains all scientific and technical information that is required to be disclosed to make the IA not misleading.

Terre A. Lane

"Terre A. Lane"

Principal Mining Engineer

Date of Signing: November 10, 2022

CERTIFICATE OF QUALIFIED PERSON

I, Richard A. Hughes, of 318 Juneau Ave., Fairbanks, AK 99701, a reviewer of the report entitled "Revised and Amended Preliminary Economic Assessment, Chandalar Mine, Chandalar Gold District" with an effective date of May 31, 2021 and an issue date of November 10, 2022 (the "IA") DO HEREBY CERTIFY THAT:

1. I am a Registered Professional Consulting Mining Engineer and have an address of 318 Juneau Ave., Fairbanks, AK 99701
2. I graduated from the University of Nevada, Mackay School of Mines with a Bachelor of Science in Mining Engineering in 1960.
3. I am a registered professional engineer in good standing in Alaska, certificate number EM-5531, and in Nevada, certificate number 2189.
4. I have traveled to and inspected the Goldrich property located on the south flank of the Brooks Range, Alaska; the visit was conducted on June 28,2019 and was a one (1) day inspection.
5. I have 61 years of experience in minerals exploration, development, operations management and engineering.
6. I have read the definition of "qualified person" set out in 17 CFR §229.1300 – (Item 1300) and certify that by reason of my education, affiliation with professional associations (as defined in 17 CFR §229.1300 – (Item 1300)) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of 17 CFR §229.1300 – (Item 1300).
7. I am responsible for reviewing and commenting on the above report to assure compliance to professional standards and to the Goldrich Mining's requirements.
8. I am a consultant to Goldrich Mining as described in section 1.5 and provide professional advice to that company on a consulting basis.
9. I have read 17 CFR §229.1300 – (Item 1300). The IA has been prepared in compliance with 17 CFR §229.1300 – (Item 1300).
10. As of the effective date of the IA, to the best of my knowledge, information and belief, the IA contains all scientific and technical information that is required to be disclosed to make the IA truthful and relevant.

Richard A. Hughes, PE

RA Hughes

Mining Engineer
Date of Signing: **November 10, 2022**

CERTIFICATE OF QUALIFIED PERSON

I, J. Larry Breckenridge, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Revised and Amended Preliminary Economic Assessment, Chandalar Mine, Chandalar Gold District" with an effective date of May 31, 2021 and an Issue date of November 10, 2022 (the "IA"), DO HEREBY CERTIFY THAT:

1. I am an Environmental Engineer and have an address at 600 Grant Street, Suite 975, Denver, Colorado 80203
2. I graduated from Dartmouth College (Bachelors of Arts, Engineering Modified with environmental Science) and the Colorado School of Mines (Masters' in Environmental Science and Engineering).
3. I am a member, in good standing, of the Board of Colorado Professional Engineers in Colorado (No. 38048), and New Hampshire (No. 12694);
4. I have 25 years of experience in environmental engineering, mine water management, and geochemistry.
5. I have read the definition of "qualified person" set out in 17 CFR §229.1300 – (Item 1300) and certify that by reason of my education, affiliation with a professional association (as defined in 17 CFR §229.1300 – (Item 1300)) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of 17 CFR §229.1300 – (Item 1300).
6. I am responsible for Sections 1.16, 1.17, 4.5, 4.6, and 20 of this IA.
7. I am independent of Goldrich Mining as described in 17 CFR §229.1300 – (Item 1300).
8. I have read 17 CFR §229.1300 – (Item 1300). The IA has been prepared in compliance with 17 CFR §229.1300 – (Item 1300).
9. As of the effective date of the IA, to the best of my knowledge, information and belief, the IA contains all scientific and technical information that is required to be disclosed to make the IA not misleading.

Larry Breckenridge

"J. Larry Breckenridge"

Principal Mining Engineer

Date of Signing: November 10, 2022