GOLD MOUNTAIN

FREEFORM CAPITAL

NI 43-101 TECHNICAL REPORT
PRELIMINARY ECONOMIC ASSESSMENT ON THE ELK GOLD PROJECT
MERRITT, BRITISH COLUMBIA, CANADA



PREPARED FOR:

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AKF MINING SERVICES INC.

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Glossary

Acronyms and Abbreviations

Ag	Silver
AKF	AKF Mining Services Inc.
Almaden	Almaden Minerals Ltd.
ALS	ALS-Chemex Laboratory
AMD	Acid Mine Drainage
ANFO	Ammonium Nitrate-Fuel Oil
AOA	Archaeology Overview Assessments
AP	Acid Potential
ARD	Acid Rock Drainage
AS	Analytic Signal
Au	Gold
BC	British Columbia
Bayshore	Bayshore Minerals Inc.
BC EAA	*British Columbia Environmental Assessment Act*
BCI	Beanstalk Capital Inc.
BD	Bacon Donaldson
BGC	Biogeoclimatic
BWi	Bond Ball Mill Work index
CABIN	Canadian Aquatic Biomonitoring Network
Caro	Caro Analytical Services
CCME	Canadian Council of Ministers of the Environment
CDN	CDN Resource Labs of Vancouver
CET	Centre for Exploration Targeting
CHG	Calculated Horizontal Gradient
CIL	Carbon-in-Leach
COC	Contaminants of Concern
COD	Contact Occurrence Density
COPC	Contaminants of Potential Concern
Cordilleran	Cordilleran Engineering Ltd.
Core6	Core6 Environmental
CV	Coefficient of Variation
EA	Environmental Assessment
Eco-Tech	Eco-Tech Laboratories Ltd.
EcoLogic	EcoLogic Consultants Ltd.
EDP	Effluent Discharge Permit
Elk Property, Elk Project, or the property	Elk Gold Project
EPCM	Engineering, Procurement, and Construction Management
EPT	Ephemeroptera, Plecoptera, Trichoptera
Equinox	Equinox Gold Corp
ERSF	East Rock Storage Facility
Fairfield	Fairfield Minerals Ltd.
FFT	Fast Fourier Transform
Freeform	Freeform Capital Partners Inc.
G&A	General and Administrative

G&T	G&T Metallurgical Services
GC	Gravity + Cyanidation
GCL	Giroux Consultants Ltd.
GF	Gravity + Flotation
GIS	Geographical Information System
GMMC	Gold Mountain Mining Corporation
GMRS	Global Mineral Resource Services
Gold Mountain or GMMC	Gold Mountain Mining Corporation
GPS	Global Positioning System
GSB	Geological Survey Branch
GSC	Geological Survey of Canada
ICP	Inductively-Coupled Plasma (Chemical Analysis Instrument)
IGRF	International Geomagnetic Reference Field
IP	Induced Polarization
ISQG	Interim Sediment Quality Guidelines
JDL	JDL Gold Corp.
K-Met	K-Met Consultants Inc.
Kinross	Kinross Gold Corp.
Knight Piésold	Knight Piésold Consulting
LG	Lerchs-Grossman
LGGC	Lions Gate Geological Consulting
LOI	Letter of Intent
LOM	Life-of-Mine
LSA	Local Study Area
MIBC	Methyl Isobutyl Carbinol
ML/ARD	Metal Leaching and Acid Rock Drainage
MOE	BC Ministry of Environment
MS	Mass Spectrometer (Analysis Instrument)
MWMP	Meteoric Water Mobile Procedure
NAG	Non-Acid Generating
NI 43-101	National Instrument 43-101
Non-PAG	Not Potentially Acid Generating
NP	Neutralization Potential
NPM	Nickel Plate Mine
NPV	Net Present Value
NSR	Net Smelter Return
O:E	Observed to Expected
OE	Orientation Entropy
PAF	Potentially Acid Forming
PAG	Potentially Acid Generating
PAH	Polycyclic Aromatic Hydrocarbons
Palmer	Palmer Environmental Consulting Group
PAX	Potassium Amyl Xanthate
PDOP	Positional Dilution of Precision
PEA	Preliminary Economic Assessment
PEL	Probable Effects Level
PET	Potential Evapotranspiration
PFS	Pre-Feasibility Study
pH	Measure of acidity or alkalinity of a solution

QA/QC	Quality Assurance and Control
QP	Qualified Person
RAAD	Remote Access to Archaeological Data
RBF	RB fault
RCP	Reclamation and Closure Plan
RH	Relative Humidity
RHR	Right-Hand Rule
RMI	Residual Magnetic Intensity
RMSF	Rock Management Storage Facility
RQD	Rock Quality Designator
RSA	Regional Study Area
RTP	Reduced to Magnetic Pole
RTP	Reduced-to-Pole
SD	Standard Deviation
SNAP	Southern Nicola Arc Project
SRM	Standard Reference Material
SWE	Snow Water Equivalent
TCexp	Total Counts—Exposure Rate
TD	Tilt Derivative
Technical Report	43-101 Technical Report Preliminary Economic Assessment on the Elk Gold Project, Merritt, British Columbia, Canada
TEM	Terrestrial Ecosystem Map
THD	Total Horizontal Derivative
TMI	Total Magnetic Intensity
Trek	Trek Mining Inc.
TSA	Timber Supply Area
UTM	Universal Transverse Mercator
VLF-EM	Very-Low-Frequency Electromagnetic
WD	Waste Dump
WQG	Water Quality Guidelines
WSC	Water Survey of Canada

Units of Measure

'	feet
"	inch
%	percent
<	less than
>	greater than
°	degree
µm	micrometre (micron)
cm	centimetre
cm^3	cubic centimetre
d	day
g	gram
g/t	grams per tonne
ha	hectare (10,000 m^2)
kg	kilogram
km	kilometre

L	litre
m	metre
M	million
m^2	square metre
m^3	cubic metre
masl	metres above sea level
mg	milligram
mL	millilitre
mm	millimetre
ms	millisecond
Mt	million tonnes
np:p	non-plant feed ratio
oz	ounce
ppm	parts per million
st	short ton (2,000 lb)
t	SI tonne (1,000 kg)

1 SUMMARY

1.1 General Statement

This report titled *43-101 Technical Report Preliminary Economic Assessment on the Elk Gold Project, Merritt, British Columbia, Canada* (the Technical Report) was prepared at the request of Freeform Capital Partners Inc. (Freeform), a British Columbia (BC) corporation listed on the TSX Venture Exchange (TSX). Freeform has entered into an agreement dated August 5, 2020 to acquire all of the issued and outstanding common shares of Bayshore Minerals Incorporated (Bayshore), a private BC corporation. Bayshore holds its interest in the Elk Gold Project (Elk Gold Project or the Project) through its wholly owned subsidiary Gold Mountain Mining Corporation (GMMC), which holds the Elk Gold Project claims and mining lease.

The purposed of this Technical Report is to provide information on the technical aspects of an exploration activity at the Elk Gold Project, to provide a PEA of the Elk Gold Project and to support listing requirements of the TSX.

This Technical Report conforms to the standards specified in National Instrument 43-101 Standards of Disclosure for Mineral Properties (NI 43-101) and Form 43-101F1.

1.2 Location, Access, and Physiography

The Elk Gold Project is located in south-central British Columbia (BC), Canada, approximately 325 km northeast of Vancouver and 55 km west of Okanagan Lake, midway between the cities of Merritt and West Kelowna. The Property is within the Similkameen Mining District and consists of 27 contiguous mineral claims and one mining lease that in total cover 16,566 ha. GMMC, a wholly owned subsidiary of Bayshore, has a 100% interest in all claims subject to a 2% net smelter return (NSR) royalty. A further 1% NSR royalty is payable on production from the Agur Option block, located approximately 4 km south of the previously mined Siwash North zone. Exploration on the Property is regulated via two permits: M-199 for mining related work on the mining lease and MX-4-387 for exploration on the surrounding claims.

Access to the Property is available via a four-lane freeway to an interchange on the northernmost claims and 2.5 km north of the historical mining area known as the Siwash North zone. A series of gravel logging roads connects to the freeway and provides access to most parts of the Property.

The Property is located within the Thompson Plateau (eastern section), in the Trepanege Plateau highland, which within the claims area consists of rolling topography ranging from 1,300 m to 1,750 m above sea level (masl). The area is blanketed by a layer of glacial till of varying thickness, and outcrop is scarce. Forest cover is mainly lodgepole pine with minor balsam, subalpine fir, and spruce. Alders are found along streams and in marshes. The claim area is about 60% clear-cut logged, and the clear-cut areas are in various stages of regrowth.

Claims which comprise the Property may be maintained by continuing to conduct work on the Property, or by cash payment in lieu. The tenure number and expiry date of each claim are listed in Table 1-1. The Mining Lease may be maintained by paying a yearly rental and providing an annual reclamation report on the Property that is acceptable to the BC Ministry of Energy, Mines & Petroleum Resources.

1.3 History

Prospecting activities in the area are said to date back to the early 1900s, and the first recorded work began in the 1960s and 1970s by several companies exploring for copper and molybdenum. Cordilleran Engineering Ltd. (Cordilleran), the exploration arm of Fairfield Minerals Ltd. (Fairfield), investigated the area for gold from 1986 to 1991, identifying and drilling nine separate zones possessing gold-mineralized quartz vein systems. Fairfield assumed operatorship in 1992 for the purpose of mining a bulk sample. Approximately 51,500 oz (1,460,000 g) of gold were produced between 1992 and 1995, mainly from a bulk sample open pit (Pit 1). That work included underground mining where drilling and limited underground test raising and stopping occurred.

Almaden Resources Corporation amalgamated with Fairfield in 2002 to form Almaden Minerals Ltd. (Almaden), which thereby became the sole owner of the Property and continued field exploration projects including drilling until late 2010.

Beanstalk Capital Inc. (BCI) purchased the Property in 2011 and changed their name to GMMC upon being listed as a publicly traded company on the TSX Venture Exchange under the symbol V.GUM. In 2012, GMMC, after two years of infill and step-out drilling, initiated a bulk sample mining operation (Pit 2). All mineralized rock mined in 2012 and 2014, plus 7,761 tonnes that remained from 1990s historical mining operations, was processed at a toll mill. The processed material had an average grade of 14.81 g/t Au, and in total, Pit 2 yielded 6,596.7 tonnes of mineralized vein material at 16.65 g/t Au from 2,443.2 m^3 of the 4,000 m^3 of banked vein permitted under Permit M-199.

GMMC merged with Lowell Copper and Anthem United in October 2016, becoming a wholly owned subsidiary of the successor company, JDL Gold Corp. (JDL). JDL merged with Luna Gold in March 2017, with JDL as the succeeding company, which subsequently changed its name to Trek Mining Inc. (Trek). In December 2017, Trek merged with NewCastle Gold Ltd. and Anfield Gold Corp., with Trek being the surviving entity. Trek then changed its name to Equinox Gold Corp (Equinox), with GMMC remaining as a wholly owned subsidiary of Equinox.

Bayshore purchased GMMC from Equinox on May 16, 2019; GMMC is now a wholly owned subsidiary of Bayshore. On August 5, 2020, Freeform and Bayshore entered into a non-binding Letter of Intent (LOI) and on August 31, 2020, entered into a definitive agreement, pursuant to which Bayshore will be acquired by and become a wholly owned subsidiary of Freeform.

1.4 Geology and Mineralization

The property is located in the Intermontane (tectonic) Belt of south-central BC. The claims are underlain by Triassic age Nicola Group volcanic-sedimentary rocks in the western third of the Property. Jurassic-aged granitic rocks of the Osprey Lake Batholith intrude the eastern portion of the Property. Feldspar-porphyry stocks and dykes of the Upper Cretaceous Otter Intrusions are mapped mainly in the southwest claim area and cut both Nicola Group volcanic rocks and Osprey Lake granitic rocks. Tertiary andesite dykes intrude all the above. Gold appears to be spatially related with the andesite dykes and is contained within pyritic quartz veins that locally cut the dykes.

Gold mineralization occurs within quartz-sulphide veins and stringers most often within phyllic- and silica-altered Osprey Lake rocks and rarely within adjacent phyllic- and silica-altered Nicola volcanic rocks. Pyrite is the most common sulphide within the quartz veins, ranging from 5% to 80%, with higher percentages often associated with chalcopyrite and tetrahedrite. Gold occurs as fine-grained free gold (typically less than 50 µm) in quartz, and also within quartz-pyrite boxworks, and in fractures within veins. Gangue minerals include quartz and altered wall rock clasts (xenoliths), with minor amounts of ankerite, calcite, barite, and fluorite. Most of the mine production in Pit 1 occurred within the quartz-monzonite and granodiorite border phase of the batholith. Mine production from Pit 2 was all from the quartz-monzonite phase.

Although nine separate vein zones have been discovered on the Property, the most significant one is known as the Siwash North zone. The Siwash North 1000 and 2000 vein complexes (historically called the B and WD Siwash Veins) are emplaced within fault/fracture zones that strike east-northeast and dip moderately to steeply southward. The veins are cut by a 3 m wide, north-northwest striking, near-vertically dipping faulted and altered mafic dyke known as the RBF. This dyke is a post-mineral structure that hosts no mineralization itself, but divides mineralized veins into two regions with little apparent offset; the veins change character across the dyke. West of the RBF the Siwash North 1300 vein is a single, moderately dipping, high-grade gold vein. East of the RBF the mineralization occurs in lower-grade multiple vein and gouge clusters.

1.5 Exploration

Exploration on the Property has progressed in a logical sequence from prospecting and geological mapping to hand soil sampling and excavator trench sampling followed by geophysical induced polarization (IP) and widely spaced drilling. Grid diamond drilling to delineate resources led to open pit and test-scale underground mining, followed by additional diamond drilling to expand the resource base and test other property targets. A second round of bulk sample mining was initiated to confirm results of the preliminary economic assessment as outlined in the 2011 Technical Report on the Property. The test mining and bulk sample program was also used to refine the mining technique and confirm parameters for full-scale operations. Airborne magnetometer and radiometric surveys were flown in early 2019 to assist in directing further property-wide exploration.

A number of the deepest holes drilled at Siwash North, particularly in the 1300 and 2500 Zones, have intersected gold mineralization greater than 5 g/t. There is insufficient drilling along strike and down dip of these intercepts to establish their limits, but given the continuity of these veins elsewhere, it is considered reasonable to anticipate that this gold mineralization may continue into areas as yet untested by drilling. Therefore, the down-dip areas of these two veins are considered to be exploration targets.

Exploration of the South, Lake, and Gold Creek Zones by trenching and drilling has encountered mineralized veins that may be similar in style to those encountered in the Siwash North Zone. To date, there is insufficient data to support a resource estimate, but the three zones are considered to be valid exploration targets.

Drilling at each of the zones has demonstrated the presence of mineralized veins over a strike length of approximately 400 m and to approximately 150 m depth down dip. The width of mineralization appears to be variable, but may range between 2 m and 5 m. Using an average bulk density of 2.7 g/cm^3, these volumes equate to a range of possible tonnages for each of the zones of between approximately 300,000 and 800,000 tonnes.

Gold assay values from the South Zone (894 assays) range from 0 g/t to 220 g/t and have an arithmetic average of approximately 0.9 g/t. Gold assay values from the Lake Zone (503 assays) range from 0 g/t to 89.7 g/t and have an arithmetic average of 1.3 g/t; those from the Gold Creek zone (251 assays) range from 0 g/t to 84.6 g/t and have an arithmetic average of 1.8 g/t.

By comparison, gold assay values from the Siwash North Zone (20,945 assays) range from 0 g/t to 1,176 g/t with an arithmetic average of 2.6 g/t. The current, pit-constrained resource estimate for the Siwash North Zone has an average gold grade of approximately 4.9 g/t at a cut-off of 0.5 g/t, or approximately twice the arithmetic average. Assuming geological similarity between these zones and the Siwash North Zone, and using the simplistic relationship between arithmetic and resource estimate average grade, the South Zone has a potential target for further exploration consisting of gold grades that range between 0.9 g/t and 1.8 g/t, the Lake Zone has a potential target for further exploration consisting of grades that range between 1.3 g/t and 2.6 g/t, and the Gold Creek Zone has a potential target for further exploration consisting of grades that range between 1.8 g/t and 3.6 g/t of gold. These ranges of possible tonnages and grades are summarized in Table 1-1.

Table 1-1: Exploration Target Potential Tonnage and Grade Ranges—Siwash North, South, Lake, and Gold Creek Zones

	Length (m)	Dip (m)	Min. Width (m)	Max. Width (m)	Min. Target (t '000s)	Max. Target (t '000s)	Min. Avg. Grade (g/t)	Max. Avg. Grade (g/t)	Min. Gold (oz '000s)	Max. Gold (oz '000s)
Gold Creek	500	150	2	5	405	1,215	1.8	3.6	23.4	140.6
Lake	500	150	2	5	405	1,215	1.3	2.6	16.9	101.6
South	500	150	2	5	405	1,215	0.9	1.8	11.7	70.3
Siwash North	500	125	2	5	338	1,013	5	10	54.3	325.5
Total					**1,553**	**4,658**	**2.1**	**4.3**	**106.3**	**638.0**

It should be noted that, for all the potential quantities and grades of these targets for further exploration, 1) the potential quantity and grade is conceptual in nature, 2) there has been insufficient exploration to define a Mineral Resource, and 3) it is uncertain if further exploration will result in the target being delineated as a Mineral Resource.

1.6 Database

The historical exploration database for the Elk Gold Project has been compiled and verified by GMMC and is supplemented by approximately 40,650 m of drilling since 2007. In all, 1,122 drill holes containing 20,552 samples were used during the estimation of resources. A relatively small portion of the area covered by the drill-hole database has been mined out during the bulk sample programs, and although the mined-out resources were not included in the resource estimate, the data were still used to assist in estimating resources in the adjacent areas.

1.7 Metallurgical Testing and Mineral Processing

A number of test programs were completed on the Elk Gold Project between 1990 and 2014. The most relevant test work on composites with head grades close to the LOM average grade was completed by G&T Metallurgical Services Ltd. (G&T) in 2008 and 2010.

The G&T Programs KM2121 and KM2522 evaluated gravity concentration with the tailings feeding, flotation, Whole-Ore Leach (WOL), and leach of the flotation concentrate. With all flowsheets the gold recovery was between 90% and 95%. The WOL circuit with gravity concentration at a primary grind in the range of P_{80} 106 µm achieved an average overall gold extraction of 95% and silver extraction of 86.5%.

Based on the testwork a pyrite-gold concentrate at the anticipated mine head grades can be produced using a flowsheet that includes gravity concentration, rougher flotation, and one stage of cleaner flotation at a primary grind size of 80% passing (P_{80}) 150 µm. The final concentrate, at an estimated mass pull between 5% and 7%, could recover approximately 90% to 95% of the gold at a grade in the range of 100 g/t using a simple reagent scheme of potassium amyl xanthate (PAX) and methyl isobutyl carbinol (MIBC).

1.8 Mineral Resource Estimation

Resources have been estimated for 14 veins that comprise the Siwash North Zone, using geological wireframe models that were developed and used for the most recent historical Mineral Resource estimate.

Because the veins extend to surface, the estimate has been constrained by a conceptual open pit. Pit parameters are: gold price US$1,361/oz; mining costs US$2.00/t; processing costs US$19.24/t; an exchange rate of C$1:US$0.74; gold selling cost of 2%; NSR royalty of 2%, overall recovery 92%; and pit slope 50°. Processing costs were based on a rate of 900 t/d.

Resources have been classified as Measured, Indicated, and Inferred according to the parameters as shown in Table 1-2. Mineral Resources that are not Mineral Reserves do not have the demonstrated economic viability. Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

Table 1-2: Elk Gold Resource Classification Criteria

Classification Category	Composites			Azimuth (°)	Dip (°)	Spin (°)	Major (m)	Median (m)	Minor (m)
	Min.	Max.	Max./Hole						
Measured	8	8	1	0	0	0	25	25	25
Indicated	4	8	1	0	0	0	50	50	50
Inferred	2	8	1	0	0	0	75	75	75

The Measured category was restricted to Zone 1300 because it is the only zone that has been explored by underground development that has provided three-dimensional (3-D) exposures of the mineralization as well as close-spaced testing by underground drilling. Resources in all the other zones were classified as Indicated or Inferred.

The Mineral Resources are tabulated in Table 1-3. The resource is divided into pit-constrained and underground portions. The pit-constrained resource is stated at a base case cut-off grade of 0.5 g/t Au, and the underground resource is stated at a base case cut-off grade of 5.0 g/t Au. Ounces and tonnes have been rounded to the nearest thousand. The Effective Date of the resource estimate is July 3, 2020.

Table 1-3: Elk Gold Mineral Resource Estimate Summary

Elk Gold Pit-Constrained Resources					
Cut-off AuEq 0.50 g/t	**Tonnes**	**AuEq (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	380,000	6.2	6.2	6.2	75,000
Indicated	2,111,000	4.8	4.7	8.5	319,000
Measured + Indicated	2,491,000	5.0	4.9	8.1	394,000
Inferred	259,000	4.6	4.5	9.6	37,000
Elk Gold Underground Resources					
Cut-off AuEq 5.0 g/t	**Tonnes**	**AuEq (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	5,000	7.48	7.42	5.70	1,000
Indicated	203,000	9.03	8.76	22.84	59,000
Measured + Indicated	208,000	8.99	8.73	22.43	60,000
Inferred	195,000	9.21	8.97	20.27	58,000

Notes: AuEq = gold equivalent
CIM definitions were followed for classification of Mineral Resources
Mineral Resources are not Mineral Reserves and have not demonstrated economic viability
Results are presented in situ and undiluted
Mineral Resources are reported at a cut-off grade of 0.5 g/t Au for pit-constrained resources and 5.0 g/t for underground resources
The number of metric tons was rounded to the nearest thousand and the number of ounces of gold to the nearest hundred.
The Resource Estimate includes both gold and silver assays. The formula used to combine the metals is: AuEq = ((AU_Cap*41.8*0.96)+(Ag_Cap*0.55*0.86))/(41.8*0.96).
The Resource estimate Effective Date is July 3, 2020.

1.9 Mineral Reserves Estimates

AKF Mining Services Inc. (AKF) has not developed a Mineral Reserve estimate for the Project as part of this PEA. Significant additional data collection and technical work are required to elevate the technical confidence of the Project to a level consistent with Mineral Reserve estimation, in accordance with the CIM *Definition Standards on Mineral Resources and Mineral Reserves,* adopted by CIM Council, as amended, NI 43-101, November 29, 2019.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources that are considered too speculative geologically have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

1.10 Mining Methods

The Elk Gold Project is envisioned to be developed as a conventional open pit mine. The operation will begin as a 70,000 t/a toll milling mine for three years. During Year 3 of operations, a 324,000 t/a mill will be constructed on site, which will commission starting in Year 4.

During Years 1 to 3, mineralized material is excavated from the open pit and placed on a limestone-capped stockpile pad. Material on the stockpile pad will be sampled and assayed for metal accounting before being shipped via highway dump trucks to a nearby mineral processing plant. Numerous potential mineral processing facilities exist within hauling distance of the Elk Gold Project.

It is anticipated that once the initial operation is underway, the owners will investigate and initiate an Environmental Assessment (EA) review, which would be required to expand the mine. This PEA envisages that by Year 4 of operations an EA Certificate will be received, and a 900 t/d processing plant will be assembled on site. The change to 900 t/d and on-site milling reduces the projected processing costs and therefore also decreases the mined cut-off grade.

Non-mineralized rock will be stockpiled in a rock management storage facility (RMSF) located west of the open pit, which is designed to have a low prominence and have 2H:1V slopes that will be capped with overburden and topsoil upon mine closure.

The total mine life will be 10 years, including processing 2.4 Mt of mineralized material grading 5.26 g/t Au and 8.78 g/t Ag. The plan includes 83.6 Mt of non-mineralized rock and an average strip ratio of 34.9 w:o.

Mining operations will be carried out by an owner-operated rental-leased fleet of mobile mining equipment including two 250 mm diameter blasthole drills, one 5 m^3 front shovel, two 10 m^3 front shovels, nine 136 tonne haul trucks, two 4 m wide blade track dozers, one 3.8 m blade rubber tire dozer, one motor grader, and one water truck.

1.11 Recovery Methods

During the initial three years of the mine life, the mineralized material will be trucked and toll treated at a nearby process facility. Starting in Year 4, the material from the mine will feed a two-stage crushing plant at a rate of 75 t/h. The process plant will be fed from the crushing circuit storage bin at a rate of 900 t/d or 42 t/h, producing a saleable gold concentrate. The process plant will operate 365 d/a at 24 h/d, with an availability of 90%.

The crushing circuit consists of a jaw crusher operating in open circuit, followed by a cone crusher operating in closed circuit, with a double-deck screen to produce a final crush size of P_{80} 10 mm.

The primary grinding circuit will consist of a single ball mill and gravity concentrator operating in reverse closed circuit with cyclones to achieve a target grind size of P_{80} 150 µm. The material will then be fed to pyrite-gold rougher/cleaner flotation circuits. The final concentrate will be sold to a facility offsite for further processing.

The tailings from the process will be dewatered and material is planned to be comingled with non-mineralized rock; thus, no tailings storage facility is contemplated for the Project.

1.12 Project Infrastructure

In the initial phase of the Project, site infrastructure is limited to the facilities required to support the surface mining operation, including modular office facilities; dry/washroom facilities; a small-scale sample processing plant for plant feed assays; hazardous waste storage area; fuel storage; Emergency Response Team facilities; and equipment and explosives storage. Power requirements at this stage are limited to powering the office facility, lighting, and fuel pumps.

In Year 4, when the 900 t/d processing facility is commissioned, additional warehouse facilities and power generation will be required to support the processing plant.

1.13 Market Studies and Contracts

The proposed project will produce a gold-sulphide concentrate which is a fungible commodity and will be saleable at the time of production.

The pit optimization carried out as a part of this PEA used a 3-year trailing average gold price of US$1,361/oz.

The economic analysis conducted as a part of this PEA used the average forward-looking gold and silver price forecasts from seven financial institutions. The average forward-looking metal prices used in the economic analysis are presented in Table 1-4.

Table 1-4: Metal Price Forecast Averages

Periods	Gold Price (US$/oz)	Silver Price (US$/oz)
2021	1,776	19.27
2022	1,717	19.07
2023	1,644	18.52
2024	1,625	18.27
Long-term	1,600	18.11

1.14 Environmental Studies, Permitting, and Social or Community Impact

Baseline environmental studies conducted at the Elk Gold Project have included: meteorology and climate; soil and rock geochemical characterization; water quantity, water quality; sediment quality; fisheries and aquatic resources; ecosystems and wildlife, archaeology; and cultural use. These studies are described individually within Section 20.

Several receptors have been identified as being potentially affected by the Project, including periphyton and aquatic plants, aquatic invertebrates, and ultimately fish (rainbow trout). Vegetation, and mammals that feed upon the plants, are also potential receptors and vectors to higher order organisms. The studies

concluded that robust management planning is needed to identify and mitigate these and other negative vectors. Working collaboratively with all potentially affected groups is desired to develop a Project plan that respects environmental and cultural needs.

Mammals that feed on vegetation and/or other animals could be negatively affected if soils are not adequately isolated from mine pollutants. Dangerous mine-created cliff edges will need to have protective berms constructed for human and wildlife safety. Human activity and interactions with wildlife in the area, whether for employment, recreation, or cultural reasons, will need to be strictly controlled. Incidents that could potentially cause injuries or death due to unplanned interaction with heavy equipment or unprotected cliff edges will need to be closely monitored and mitigated.

Several mitigations to potential environmental effects were suggested by the study. Prompt removal of mineralized material from site will ensure acid rock drainage does not lead to metal leaching and environmental contamination. Ninety percent of mined non-mineralized rock is non-acid generating (NAG); however, careful management of mining NAG rock is essential to maintaining healthy hydrologic and hydrogeologic environments. Vegetation supported by various ecosystems on the Property that could potentially be impacted by the Project could be mitigated by careful planning and sequencing of mine operations. Invasive plants introduced or allowed a foothold in disturbed areas which could threaten local vegetation need to be controlled or eradicated.

Bayshore has been engaging with 26 First Nations bands and associations in conjunction with the submission of a 70,000 t/a mining and effluent discharge permit amendment in May 2020. Of the 26 indigenous groups, nine are participating in the Mine Review Committee related to that application. In parallel to the permit application consultation, Bayshore is actively negotiating Impact Benefit Agreements with several First Nations.

1.15 Capital and Operating Costs

Total life-of-mine (LOM) capital cost is anticipated to be $60.3 million. This includes capital costs for the operation and associated owners costs. Initial mining capital costs are estimated to be $6.9 million, which accounts for the fact that the bulk of mobile equipment is planned to be leased over the LOM. As equipment will be rental-leased, there is no planned replacement capital costs for mining equipment. The total LOM mining capital cost is $9.2 million.

The processing plant capital cost is incurred in Year 3 of the mine plan, prior to commissioning of the plant in Year 4. The estimated capital cost for purchase and assembly of the modular processing plant is $23.8 million. An allowance of $200,000 per year of sustaining capital is included.

Reclamation and closure activities are estimated to cost $10 million, and that cost is spread over two years at the end of the mine life.

Mine operating costs were developed based on planned equipment productivities, fuel consumption, and maintenance requirements. The LOM operating cost for the mine is $255 million, which equates to $3.03/t mined or $109/t milled.

The toll milling operating cost for Years 1 to 3 of the mine plan is estimated at $45/t milled, based on the known operating costs of potential toll milling partners in the project area, and increasing those costs by a factor of three to account for the mill operators' margin.

When the on-site 900 t/d mill is commissioned, the operating cost of the mill is estimated to be $21.03/t milled.

General and administrative (G&A) costs include administrative staff, Mine Rescue costs, and associated supplies. The total G&A cost is estimated to be $1.6 million per year, which equates to $4.96/t milled.

1.16 Economic Analysis

The economic analysis presented herein is preliminary in nature, includes Inferred resources that are considered too speculative geologically to have economic value place on them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. There is no guarantee that Inferred resources will convert to Indicated or Measured resources in the future.

A PEA-level economic model was developed for the Elk Gold Project using a long-term gold price of US$1,600/oz, which is the average of seven financial institution's long-term price forecasts. The economic model includes the LOM production of 373,000 oz Au and 609,000 oz Ag, with a total revenue net of selling costs of $788 million. This results in a pre-tax net present value (NPV) (5%) of $318 million and an after-tax NPV (5%) of $191 million.

This economic analysis indicates that the Elk Gold Project is a potentially economic, conventional open pit mining operation with an after-tax NPV (5%) of $191 million.

Sensitivity analysis indicates that to achieve a breakeven after-tax NPV the price of gold and silver would have to drop by approximately 49% to $784/oz Au and $9/oz Ag. Conversely, total cash costs would have to increase 68% to achieve a break-even after-tax NPV. Table 1-5 presents a sensitivity analysis.

Table 1-5: Effect of Variation in Gold Price on Project NPV

Long-Term Gold Price (US$/oz)	Pre-Tax NPV (5%) ($ million)	After-Tax NPV (5%) ($ million)
2,200	543.2	325.2
2,000	468.2	280.4
1,800	393.3	235.7
1,600	**318.3**	**190.9**
1,400	243.3	144.9
1,200	168.2	97.1

1.17 Interpretations, Conclusions, and Recommendations

As with most projects, many risks could affect the economic outcome. The significant external risks to the Elk Gold Project are gold prices, First Nations support, exchange rates, transport costs, smelter terms, government regulations, taxes, shareholder support, and the availability to raise financial resources.

Opportunities exist that could improve the potential of the project. Most of these opportunities are also potential risks, as explained above. For example, pit slope angles present both a risk and opportunity.

The authors recommend the following for Phase I and Phase II programs:

Phase I

- Drill program targeting the resources in the Siwash North zone to test and expand the modelled mineralization.
- In-fill and step out drill program targeting the Lake Zone with the objective of declaring an Inferred Resource.

Phase II

- Review the existing geological model wireframes to optimize the drill-hole intercepts and consistency of geological interpretation.
- Complete a pre-feasibility study (PFS).
- Conduct additional metallurgical testwork to optimize the project flowsheet and to assess variability in the plant feed.
- Follow-up on airborne geophysical survey results to determine priority targets for further testing.
- Complete a series of exploratory diamond drill holes in the Elusive Creek zone to better develop that exploration target.
- In-fill and step-out drilling in the South Showing and Gold Creek Zone with the objective of declaring an Inferred Resource.
- Include geotechnical drilling to confirm appropriate slope angles for future open pit design work.
- The authors estimate the total cost of this work to be $2.565 million. Table 1-6 shows the proposed programs and costs.

Table 1-6: Recommended Two-Phase Program and Estimated Costs

Item	Cost ($ million)
Phase I Program	
Siwash North Resource Drill Program	0.350
Lake Zone Drill Program	0.150
Subtotal of Phase I	*0.500*
Phase II Program	
Review Wireframe Model	0.020
Complete an PFS Study	0.250
Metallurgical Testwork	0.350
Geophysical Analysis	0.035
Elusive Creek Drill Program	0.150
Gold Creek Drill Program	0.650
South Showing Drill Program	0.495
Geotechnical Drilling	0.150
Subtotal of Phase II	*2.100*
Total	**2.600**

2 INTRODUCTION AND TERMS OF REFERENCE

2.1 Introduction

This Technical Report presents the results of an airborne geophysics survey, a resource review and re-estimation, and a PEA of the Elk Gold Project.

2.2 Terms of Reference

This Technical Report has been prepared for Freeform by Antonio Loschiavo, P.Eng., Robert G. Wilson, P.Geo.; Gregory Z. Mosher, P.Eng.; and Kelly McLeod, P.Eng.; the Qualified Persons (QP) for the preparation of this Technical Report, as defined in National Instrument (NI) 43-101. This report documents an airborne geophysical survey and the re-estimation of resources on the Property, near Merritt, BC, as of July 3, 2020.

Currencies outlined in this Technical Report are stated in Canadian dollars unless otherwise stated.

2.3 Report Authors

Table 2-1 provides a list of the QPs involved in preparing this Technical Report.

Table 2-1: List of Qualified Persons

Name	Company	Qualifications and Affiliations	Sections Authored	Personal Inspection
Antonio Loschiavo	AKF Mining Services Inc.	P.Eng. (BC)	Sections 1.1, 1.9, 1.10, 1.12, 1.13, 1.15, 1.16, 1.17, 15, 16, 18 to 22, 25.5, 26.1, 26.2.2, 26.2.8, 26.2.9, and 27	Multiple visits from 2012 to May 14, 2020
Robert G. Wilson	RGW Geoscience	P.Geo. (BC)	Sections 1.2 to 1.4, 1.6, 1.14, 2 to 12 (except 10.4), 23, 24, 25.1, 25.2, 25.4, and 26.2.4 to 26.2.7	Multiple visits yearly from 2011 to May 14, 2020
Gregory Z. Mosher	GMRS	P.Geo. (BC)	Sections 1.5, 1.8, 10.4, 14, 25.3, and 26.2.1	June 21, 2019
Kelly MacLeod	K-Met Consultants Inc.	P.Eng. (BC)	Sections 1.7, 1.11, 13, 17 and 26.2.3	Did not conduct a site visit

3 RELIANCE ON OTHER EXPERTS

The authors have relied on Freeform for information pertaining to Property legal description, ownership, permitting, and legal obligations pertaining to purchase and maintenance of the Property. Information pertaining to legal and ownership aspects of the Property were obtained from Freeform on August 23, 2019 in the form of letters of opinion by Blake, Cassels & Graydon LLP dated May 16, 2019, and by Kenneth Adessky of Adessky Attorneys dated August 23, 2019.

4 PROPERTY DESCRIPTION AND LOCATION

The Elk Gold Project is located in south-central BC, approximately 325 km northeast of Vancouver and 55 km west of Okanagan Lake, midway between the cities of Merritt and West Kelowna (Figure 4-1). The Property is within the Similkameen Mining District and consists of 27 contiguous mineral claims covering 16,716 ha and one mining lease covering 150 ha (Figure 4-2). GMMC has a 100% interest in all claims and the lease, subject to a 2% NSR royalty to Almaden. A further 1% NSR royalty is payable to Don Agur on production from the Agur Option block, located approximately 4 km south of the previously mined Siwash North zone.

In May 2019, Bayshore completed a purchase agreement for the Elk Gold Project with Equinox Gold. Under the terms of Bayshore's purchase agreement, Equinox Gold completed sale of the Property by way of Bayshore's purchase of the shares of Equinox Gold's wholly owned subsidiary, GMMC. The specific terms include $1 million in cash on closing of the sale; and $9 million in a first ranking secured promissory note (the (Promissory Note), payable in annual installments of $3 million commencing two years from closing). The total amount due under the Promissory Note may be adjusted such that if Freeform pays a total of $8 million within two years from closing, that will represent full and final payment; or, if Freeform pays a total of $8.5 million within three years from closing, that will represent full and final payment.

In August 5, 2020, Freeform entered into a non-binding Letter of Intent (LOI) with Bayshore to acquire 100% of the outstanding Bayshore shares and to acquire GMMC.

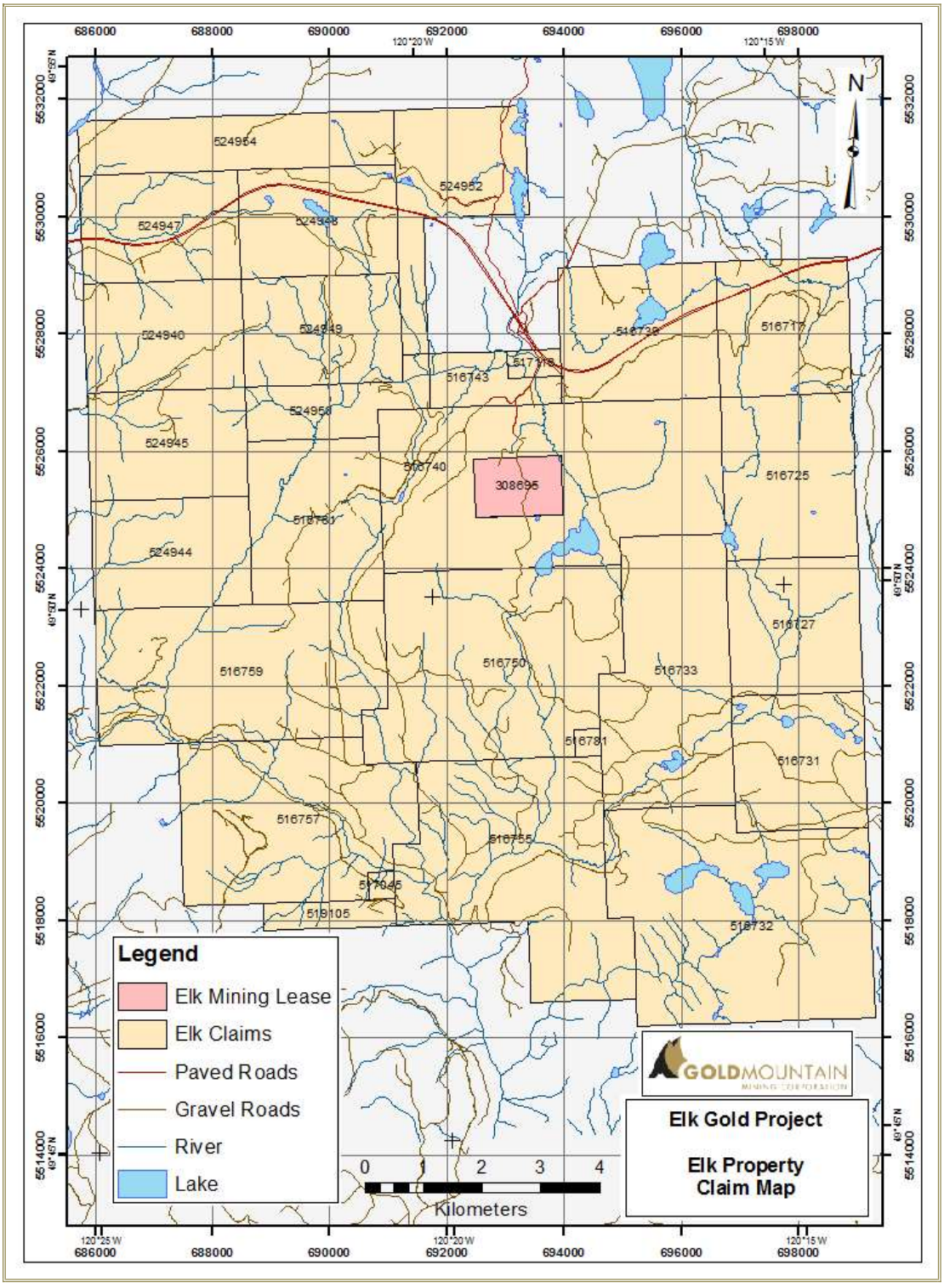
Bulk sample mining operations on the mining lease are permitted under *Mines Act* Permit M-199, first issued in 1995 and as amended in 2012. Reclamation bonding posted by Bayshore under M-199 in the amount of $150,000 is held by the government. Exploration on the Property is regulated via two permits: M-199 for work on the mining lease and MX-4-387 for exploration on the surrounding claims. Reclamation bonding posted by Bayshore under MX-4-387 in the amount of $10,000 is also held by the government. Although reclamation has been ongoing throughout the life of the Property, 12.51 ha out of a total disturbance area of 42.61 ha remains to be reclaimed, most of which is attributed to access roads, pit surrounds, and rock-plus-soil storage facilities. The reader is cautioned that reclamation is only considered to be complete when the government determines that all reclamation has been carried out to satisfactory standards, and has achieved the Property's end land-use plan goals.

Claims comprising the Property are listed in Table 4-1 along with the tenure number, tenure type, size in hectares and expiry date of each claim. The claims may be maintained beyond their current expiry date by continuing to conduct work on the Property at the rate of $331,321/a, or by cash payment in lieu at double that rate. The Mining Lease may be maintained by paying a yearly rental of $3,000 and providing an annual reclamation report on the Property that is acceptable to the Ministry of Energy, Mines & Petroleum Resources.



Source: Bayshore, August 2020.

Figure 4-1: Elk Gold Project Regional Location



Source: Bayshore, August 2020.

Figure 4-2: Elk Gold Project Claim Map

Table 4-1: List of Claims

Tenure Number	Claim Name	Tenure Type	Issue Date	Expiry	Area (ha)
308695		Lease	1992/sep/14	2021/sep/14	150.0
516717		Claim	2005/jul/11	2021/apr/30	520.568
516725		Claim	2005/jul/11	2021/apr/30	624.976
516727		Claim	2005/jul/11	2021/apr/30	521.047
516731		Claim	2005/jul/11	2021/apr/30	521.261
516732		Claim	2005/jul/11	2021/apr/30	1,481.067
516733		Claim	2005/jul/11	2021/apr/30	938.034
516739		Claim	2005/jul/11	2021/apr/30	624.689
516740		Claim	2005/jul/11	2021/apr/30	1,458.278
516743		Claim	2005/jul/11	2021/apr/30	166.607
516750		Claim	2005/jul/11	2021/apr/30	1,271.493
516755		Claim	2005/jul/11	2021/apr/30	1,188.843
516757		Claim	2005/jul/11	2021/apr/30	1,021.841
516759		Claim	2005/jul/11	2021/apr/30	1,125.588
516761		Claim	2005/jul/11	2021/apr/30	625.034
516781	ELK05A	Claim	2005/jul/11	2021/apr/30	20.849
517045		Claim	2005/jul/12	2021/apr/30	20.858
517116	ELK05B	Claim	2005/jul/12	2021/apr/30	41.65
519105		Claim	2005/aug/16	2021/apr/30	104.301
524944	ELK06A	Claim	2006/jan/09	2021/apr/30	500.072
524945	ELK06B	Claim	2006/jan/09	2021/apr/30	499.901
524946	ELK06C	Claim	2006/jan/09	2021/apr/30	499.729
524947	ELK06D	Claim	2006/jan/09	2021/apr/30	499.558
524948	ELK06E	Claim	2006/jan/09	2021/apr/30	499.556
524949	ELK06F	Claim	2006/jan/09	2021/apr/30	499.727
524950	ELK06G	Claim	2006/jan/09	2021/apr/30	270.751
524952	ELK06H	Claim	2006/jan/09	2021/apr/30	520.326
524954	ELK06I	Claim	2006/jan/09	2021/apr/30	499.429

The BC Ministry of Environment (MOE) issued a Discharge Permit (Permit 106262) October 6, 2014, as amended July 30, 2015, under the Provisions of the *Environmental Management Act*. As per the Permit conditions, the permit is maintained by monitoring water quality of any discharge and by reporting the findings in an annual report to the MOE.

Execution of the PEA mine plan proposed herein will require an amendment to the existing Mine Permit and Effluent Discharge permits. Bayshore submitted a joint mine permit amendment application in May 2020 which will permit the development of a 70,000t/a mine plan using a toll mill. This is in line with Years 1 to 3 of the proposed mine plan.

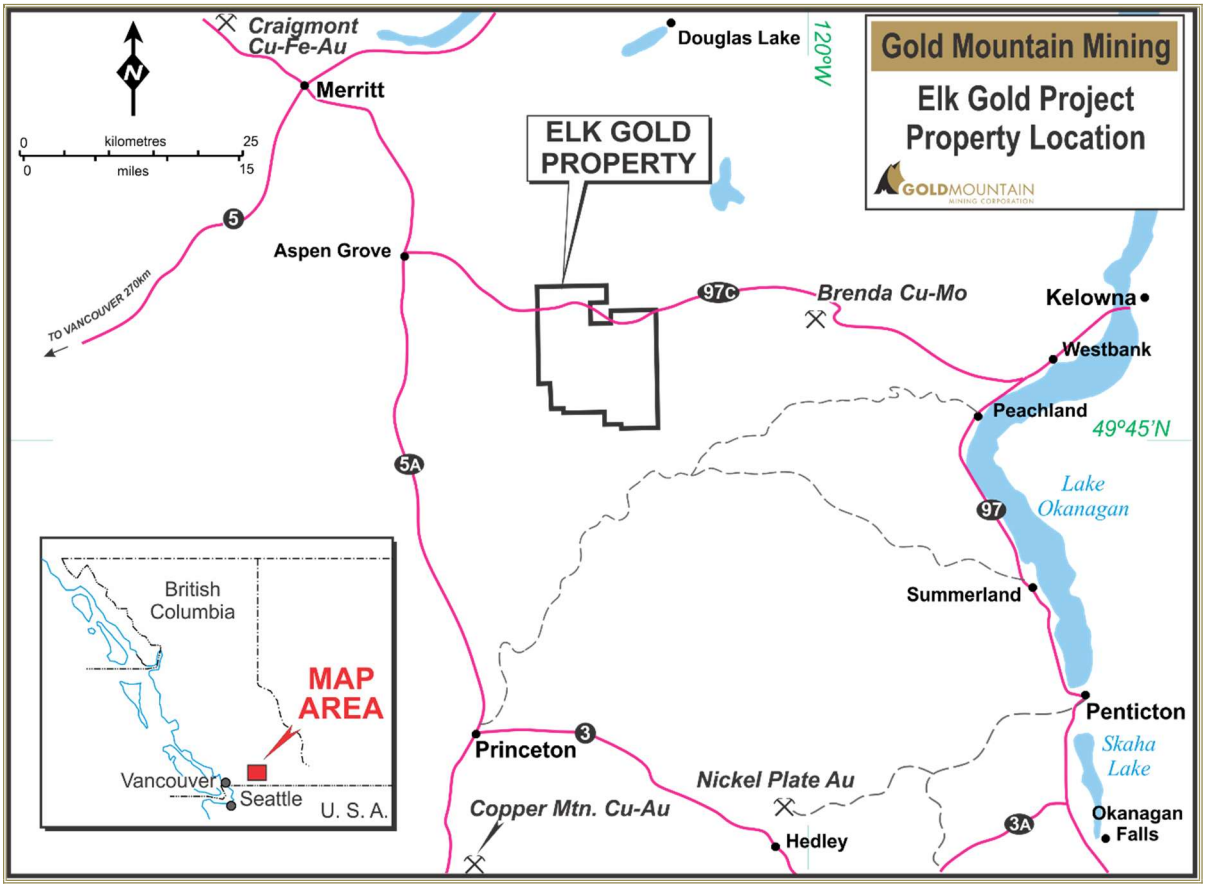
Starting in Year 4 of this mine plan, the mine will expand to 900 t/d. It is anticipated that this will require an EA Certificate and amendment to the mining and effluent discharge permits.

Additional diamond drilling on the project will require a Notice of Work under the existing Exploration permit M-4-387 for drilling on the mineral claims. Diamond drilling on the mining lease may be authorized via a Departure from Approval on the mining permit M-199.

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

5.1 Accessibility

Access to the Property is obtained by following the Okanagan Connector (Highway 97C) east from Merritt for 50 km to the Elkhart Road interchange. The historical Siwash North open pit is located 2.5 km south of the interchange. If approaching from the east, the same highway would be followed 50 km west from West Kelowna (formerly Westbank) (Figure 5-1). The highway passes through the Property's northernmost claims. From the Elkhart Road interchange on Highway 97C, gravel roads and trails provide access to most parts of the Property for all sizes of vehicles.



Source: Bayshore, August 2020.

Figure 5-1: Elk Gold Project Location

5.2 Climate

Environment Canada weather data collected over the last 30 years from stations near the Property show that it has relatively short, warm summers, and cold winters. Daily temperatures for May through September range from 12°C to 18°C, with extremes of 5°C to 39°C. Average daily temperatures during the months of November to February are 0°C to −5°C, with extremes ranging from 14°C to −9°C. The average yearly rainfall is 235 mm, with most of it falling from May to October. Monthly rainfall averages are between 20 mm and 25 mm, with June being the wettest month at 34 mm. The yearly snowfall average is 83 cm, with most of it falling between December and February. The effect that winter conditions may have on potential commercial mining operations is beyond the scope of this Technical Report; however, this PEA envisages year-round operations which is common for surface mining operations around BC.

5.3 Local Resources

The main industries within the area are cattle ranching, logging, and recreational tourism, with fishing available on small lakes across the Plateau. The once-dense forest cover supported hunting of deer, moose, and game birds (Wildstone Resources, 1993), though with logging, much of the forest has become fragmented.

Most of any needed supplies or services can be sourced from the cities of Kelowna, West Kelowna, Kamloops, and Merritt. All other needs may be obtained from Metro Vancouver or cities within the Fraser Valley, a four-hour drive to the west. West Kelowna is the current location for equipment storage space for the Property.

5.4 Infrastructure

The Property is serviced by a four-lane freeway with an interchange 2.5 km north of the Siwash North zone. All-weather logging roads provide access to all parts of the Property. Surrounding the mining lease, GMMC holds a large land area under claim. Surface rights are currently held by the government. Permits under the provincial *Water Sustainability Act* would have to obtained for mining-related water use.

A single-phase power line ends at Elkhart Lodge, 2 km from the mine lease. Three-phase power is located approximately 20 km east and west of the Project site. For the purposes of this PEA, diesel generated power is planned to support the mine plan. Connecting the site to grid power is an opportunity that should be investigated in the future.

Kelowna, West Kelowna, and Peachland are 62 km east of the Property, have an international airport, and have, to date, been an adequate source of labour, supplies, and equipment. Merritt is approximately 40 km west of the Property and is a potential community for a commuting workforce.

Existing on-site infrastructure includes a 20-person exploration camp that consists of five wood tent-frame sleeping quarters, four wood foundations for aluminum-framed Jutland tents, a first-aid tent frame, a dry building that includes showers and sinks, a generator shack, a kitchen and dining room building, as well as a water pump/filtration system building, the last four having aluminum roofs. All facilities were cleaned out and closed after the 2012 exploration program. Lodging for employees and contractors was off site during the 2013 and 2014 seasons. This PEA assumes that all personnel will live in the surrounding communities and commute to the Property.

The exploration office area consists of three large metal storage containers, a core shack along with a core-splitting shack, an office building, and core racks for long-term storage of core. In fall 2014, vandalism caused damage to the office and core shack, which were repaired prior to closing. Several core racks were also destroyed with the loss of geological knowledge. Ongoing theft has resulted in loss of much of the on-site equipment.

The crushing and sampling area comprises a semi-stationary crushing unit (sample plant, which has temporarily been relocated to off-site storage) and three 20 ft metal storage containers. An aluminum framed tent with wooden foundation was erected during mining to provide an area where samples from all stockpiles could be laid out and dried. Although the tent was dismantled, the plywood floor was left for future programs.

The mining lay-down had one ATCO trailer used as an office. The trailer was removed at the end of the 2014 season. Explosives magazines, which were emptied at the close of mining, were removed in 2017. A silo with screw loader, designed for ANFO storage, is on site near the mining lay-down area. The silo was never used, contains no explosives, was left on site after the 2014 season, and remains there. A small storage unit in West Kelowna contains miscellaneous exploration equipment such as first aid gear, core-cutting saws, pumps, generators, and water filtration equipment.

5.5 Physiography

The Property is located within the Thompson Plateau (eastern section) known as the Trepanege Plateau highland, which, within the claims, area consists of rolling topography ranging in elevation from 1,300 masl to 1,750 masl. The area is blanketed by a layer of glacial till of varying thickness, and outcrop is scarce. Forest cover is mainly lodgepole pine with minor balsam, subalpine fir, and spruce. Alders are found along streams and in marshes. The entire claim area is about 60% clear-cut logged.

6 HISTORY

Details of the Property history are drawn from a number of sources, including King (2001), Jakubowski (2006), and Giroux (2007). While prospecting activities in the area date back to the early 1900s, the first recorded work began in the 1960s and 1970s by several companies exploring for copper and molybdenum. Cordilleran, the exploration arm of Fairfield, investigated the area for gold from 1986 to 1991. Fairfield assumed operatorship in 1992 for the purpose of mining a bulk sample. Approximately 1,460,000 g (51,500 oz) of gold were produced between 1992 and 1995 (Blythe, 2007) mainly from a bulk sample open pit (Pit 1). That work included underground mining where a decline was driven to create drill stations, and limited underground test raising and stopping occurred (Rowe, 1995).

Almaden Resources Corporation amalgamated with Fairfield Minerals Ltd. (Fairfield) in 2002 to form Almaden Minerals Ltd. (Almaden), which thereby became the Property's sole owner, and continued field exploration projects until late 2010. In April 2007, GCL estimated a resource (Giroux, 2007), which included 2005 diamond drill-hole data. In 2010, Almaden received an updated geological model and NI 43-101 Mineral Resource estimates from Lions Gate Geological Consulting Inc. (LGGC) (LGGC, 2010). LGGC's new model and resource estimate incorporated 2007 drilling results that were not included in Giroux's (2007) previous estimate. The new model recognized eight separate vein zones that comprise the B (now 1000 series) vein system and four separate zones that comprise the WD (now 2000 series) veins. The 2010 Mineral Resource estimate for the Property was estimated using 0.5 g/t Au cut-off for blocks that are within the resource estimation pit shell and a 5.0 g/t Au cut-off for blocks below the pit shell (Table 6-1). That resource estimate was 200,000 tonnes at 8.77 g/t Au Measured, 2.0 Mt at 3.82 g/t Indicated, plus 1.15 Mt at 7.13 g/t Au Inferred.

Table 6-1: Mineral Resources to 2007 Drilling for the 1000 and 2000 Series Veins

Vein	Location	Class	Cut-off Au (g/t)	Tonnage	Au (g/t)	Ounces
B&WD	In Pit	Measured	0.50	200,000	8.77	55,000
B&WD	In Pit	Indicted	0.50	1,870,000	3.50	210,000
B&WD	In Pit	M&I	0.50	2,070,000	4.00	266,000
B&WD	In Pit	Inferred	0.50	640,000	5.71	117,000
B&WD	Below Pit	Measured	5.00	-	-	-
B&WD	Below Pit	Indicated	5.00	130,000	8.46	35,000
B&WD	Below Pit	M&I	5.00	130,000	8.46	35,000
B&WD	Below Pit	Inferred	5.00	510,000	8.91	146,000
B&WD	In & Below Pit	Measured	0.50 & 5.00	200,000	8.77	55,000
B&WD	In & Below Pit	Indicated	0.50 & 5.00	2,000,000	3.82	245,000
B&WD	In & Below Pit	M&I	0.50 & 5.00	2,190,000	4.26	301,000
B&WD	In & Below Pit	Inferred	0.50 & 5.00	1,150,000	7.13	263,000

The most recent resource estimate was carried out in 2016. This estimate was compliant with NI 43-101 but is no longer current because of changes in ownership of the Property. No additional exploration has been conducted on Siwash North since the completion of the 2016 estimate.

Resources were estimated for the Siwash North 1000 (B) and 2000 (WD) vein complexes. Grade continuity was quantified by semivariogram analysis with the directions and distances of maximum gold continuity determined for each vein.

Classification of individual kriged blocks was based on a measure of the grade continuity, namely the semivariograms. Blocks estimated during Pass 1, with search ellipse limited to ¼ the semivariogram range, were classed Measured in areas with sufficient drill-hole density. Those estimated during Pass 2 at ½ the range were classed as Indicated. All others were classed as Inferred.

The potential open pit resources were evaluated based on constraining pit shell parameters of US$1,232/troy ounce, a US$:C$ exchange rate of 0.84, a plant processing rate of 500 t/d and an average gold recovery of 92%.

From the above parameters, any blocks located within the constrained shell were reported using a 1.0 g/t cut-off and any blocks below the pit shell surface was reported using a 5.0 g/t cut-off.

The results are presented as grade-tonnage tables for the mineralized portion of the blocks (Table 6-2). In other words, the resource available if one could mine to the 3D Solid boundaries.

The resource is summarized at 1.0 g/t cut-off for the open pit, which includes all blocks within the pit shell and at a 5.0 g/t cut-off for the underground which includes all blocks below the pit that might be accessible to underground extraction. Results of this estimate are presented in Table 6-2.

Table 6-2: Siwash North Mineral Resource Estimate 2016

Location	Class	Cut-Off Au g/t	Tonnage	Au g/t	Ounces
In Pit	Measured	1.0	340,000	7.07	77,300
In Pit	Indicated	1.0	691,000	5.83	129,500
In Pit	M&I	1.0	1,031,000	6.24	206,800
In Pit	Inferred	1.0	823,000	4.56	120,800
Below Pit	Measured	5.0	-	-	-
Below Pit	Indicated	5.0	11,600	13.73	5,100
Below Pit	M&I	5.0	11,600	13.73	5,100
Below Pit	Inferred	5.0	273,900	10.09	88,800
In & Below Pit	Measured	1.0 & 5.0	340,000	7.07	77,300
In & Below Pit	Indicated	1.0 & 5.0	702,600	5.96	134,600
In & Below Pit	M&I	1.0 & 5.0	1,042,600	6.32	211,900
In & Below Pit	Inferred	1.0 & 5.0	1,096,900	5.94	209,600

BCI purchased the Property in 2011 and changed their name to GMMC upon being listed as a publicly traded company on the TSX Venture Exchange under the symbol V.GUM. GMMC owned 100% of the property subject to a 2% royalty payable to Almaden. A further 1% NSR is payable to Don Augur on production from the Agur Option block, located approximately 4 km south of Siwash North Zone. In 2015 Almaden spun its royalty out to newly formed company Almadex Minerals Ltd., which in 2018 split into two companies with the royalty transferred to a numbered company that changed its name to Almadex Minerals Ltd.

In 2012, after two years of infill and step-out drilling, GMMC initiated a bulk sample mining operation, the second on the Property (Pit 2). All mineralized rock mined in 2012 and 2014 plus that remaining from 1990s historical mining operations totalling 7,761 tonnes at an average grade of 14.81 g/t Au, was processed at the Kettle River Mill owned by Echo Bay Minerals Company, a subsidiary of Kinross Gold Corp. (Kinross), near Republic, Washington, U.S.A. In total, Pit 2 yielded 6,596.7 tonnes of mineralized vein material at 16.65 g/t Au from 2,443.2 m^3 of the 4,000 banked m^3 of vein permitted under *Mines Act* Permit M-199. Mining details are discussed in Section 6.2.

GMMC merged with Lowell Copper and Anthem United in October 2016, becoming a wholly owned subsidiary of the successor company JDL. JDL merged with Luna Gold in March 2017, with JDL as the succeeding company, which subsequently changed its name to Trek. In December 2017 Trek merged with NewCastle Gold Ltd. and Anfield Gold Corp., with Trek being the surviving entity. Trek then changed its name to Equinox, with GMMC remaining as a wholly owned subsidiary of Equinox.

Bayshore purchased Equinox's shares in GMMC on May 16, 2019, and GMMC became wholly owned subsidiary of Bayshore. As discussed in Section 4, the terms of Bayshore's purchase agreement include $1 million in cash on closing of the sale; and $9 million in the Promissory Note payable in annual installments of $3 million, commencing two years from closing. The total amount due under the Promissory Note may be adjusted such that if Bayshore pays a total of $8 million within two years from closing, that will represent full and final payment; or if Bayshore pays a total of $9.5 million within three years from closing, that will represent full and final payment.

On August 5, 2020, Freeform and Bayshore entered into a non-binding LOI to enter into a definitive amalgamation agreement pursuant to which Bayshore will become a wholly owned subsidiary of Freeform.

6.1 Exploration History

In the 1960s and early 1970s the Elk Gold Project area was explored for gold, copper, and molybdenum with little or no success. A summary of those programs is as follows:

- 1955–1995—Don Agur, of Summerland, BC, prospected the north and west parts of the present Property and to the south along Siwash Creek.

- 1972—Phelps Dodge Corporation of Canada, Ltd., conducted mapping and soil geochemistry in search of copper mineralization in the present claim tenure #516759 and #516757.
- Utah Mines Ltd. conducted mapping, geochemistry, induced polarization (IP) geophysics, and trenching in search of copper mineralization on part of what is now tenure #516759.
- 1979–1981—Brenda Mines Ltd. conducted copper exploration including mapping, soil geochemistry, geophysics, trenching, and diamond drilling. Work was done in the southern part of the Property, currently covered by tenure #s 516755, 516757, and 516759.
- 1980—Cominco Ltd. conducted geological mapping and soil geochemistry in search of molybdenum mineralization. The work was completed in current tenure #s 515727, 516731, 516733, and 516740.

The 1980s saw resurgence in regional exploration around the Property, this time for its gold potential. Placer gold in creeks surrounding the Property encouraged prospecting within newly established logging areas. In 1986, gold- and sulphide-bearing quartz vein float found at the Discovery zone led to the initial Elk Gold Project claims being staked by Cordilleran for Fairfield.

Cordilleran conducted the exploration from 1986 to 1991, and Fairfield from 1992 to 2001. As mentioned above, in 2002, following Fairfield's merger with Almaden Resources Corporation, Almaden completed exploration between 2002 and 2010. GMMC purchased the Property in 2011 and assumed operatorship that year.

From 1987 to 1989 Cordilleran completed detailed grid soil sampling, trenching, road building, geophysical surveying, and diamond drilling over the Siwash North and Siwash Lake areas. These surveys identified several roughly east–west trending zones containing gold mineralization which, from north to south over 6 km, were named Bullion, Siwash North, Gold Creek, Lake, Great Wall/End, Discovery, Elusive Creek, and South Showing.

In 1990 and 1991, the Siwash North zone was identified as the most promising zone for containing significant gold quantities and was clear-cut logged to facilitate the zone's development. Surface grid drilling, open pit bulk sampling and underground fan drilling dominated the period from 1992 to 1994, after which exploration was scaled back, with small drill programs occurring in 1995 and 1996. Following intensive reclamation in 1997 the Property was not active in 1998 and 1999. During the 2000 field season, a Fairfield diamond-drill program tested additional vein systems, and a limited trenching program was performed in 2001. Almaden conducted drill programs from 2002 to 2007. The Property was inactive in 2008 and 2009, but Almaden restarted drilling in 2010. In 2011 GMMC completed diamond drilling on the Siwash Zone, as well as Gold Creek, Lake, Far West, and South zones. Pole-dipole IP and Titan 24 geophysical surveys were also completed that year. Additional drilling by GMMC in 2012 further tested the Siwash North, Bullion, Nicola, Lake, Great Wall, Discovery, and South Showing zones, and completed the IP surveys first started in 2009.

Exploration on the Property has progressed in a logical sequence from prospecting and geological mapping to hand soil sampling and excavator trench sampling, followed by geophysical IP and

magnetometer surveys, and widely spaced drilling. Grid diamond drilling to delineate resources led to open pit and test-scale underground mining followed by additional diamond drilling to expand the resource base and test other Property targets.

A full description of exploration work is documented in yearly assessment reports authored by Jakubowski (1987–1991, 1995, 1999–2000, 2002–2006) as well as by King (2001) and Giroux (2007). Recent exploration work has been detailed in unpublished reports by Alexander (2011), Benoit, Wilson, & Daubeny (2012), Benoit & Wilson (2013), and Benoit, Sanders, & Wilson (2014). These reports are the source for data presented in this section. A summary of all exploration since 1986, not including drilling, is listed in Table 6-3.

Table 6-3: Summary of Exploration other than Drilling

Geochemical Soil Surveys	1986–1996	17,400 Samples Collected
Litho Geochemical Samples	1987–1992	5,728 samples collected
	2012	16 samples collected
	2013	57 samples collected
Metallurgical Samples	1990, 2007, 2014	Samples collected and sent for metallurgical testwork
Geological Mapping	1997–98,1990,1992	4,110 ha mapped
	2013	105.5 ha mapped
Trenches	1986–1996	7,905 m
	2001	202 m
	2004	20 m
Road Building	1987–88, 1990–1991	14.9 km
	2004	500 m
	2005	350 m
Legal Surveys	1990–92	1,824 km
Geophysical Surveys	1987,1989,1990	Mag.: 105 km, IP 4.5 km
	2004,2009	Mag.: 15.8 km
	992	UTEM: 1.8 km
	2009 –2011	IP: 38 km
	2011	Titan24: 12 km
	2012	IP: 35.2 km
Underground Development	1993–1994	Portal and decline driven 995 m
Open Pit Bulk Sampling & Limited Mining	1992–1994	480,000 m^3 waste 16,230 tonnes mined; 47,600 oz Au produced
	2012–2014	Bulk Sample, 38,582 m^3 soil, 166,739 m^3 waste, 2,443.3 m^3 vein material mined
Reclamation	1989–1997, 2011	~15 ha
Mining Disturbance	2012–2016	1.49 ha

Source: King (2001), Giroux (2007), Jakubowski (2002–2007), Benoit, et al. (2011–2014)

The primary exploration activity post mining has been diamond drilling from surface, with most drill testing on the Siwash North and three other zones. A list of all Elk Gold Project exploration activities

completed by Cordilleran, Fairfield, Almaden, and GMMC, including drilling in the B and WD Vein Complex (renamed to 1000 and 2000 series veins) is shown below (Alexander, 2011; Benoit, Saunders, & Wilson, 2014; Benoit & Wilson, 2012, 2013; Jakubowski, 2006; Wong & Blythe, 2008, 2009, 2010):

- 1987—Soil geochemistry, geophysical surveys, and trenching identified gold-bearing quartz vein structures extending for hundreds of metres.
- 1988—Trenching in B and WD Vein Complex (historically called the Siwash North Area) and Elusive Creek area.
- 1989—Trenching and stripping of overburden exposed quartz veins, which extended the gold-bearing vein system 750 m along strike, and 12 HQ-sized diamond drill holes (778 m) were completed to confirm down-dip continuity in the B and WD Vein Complex. Geophysical surveys and soil sampling were done in other areas of the Property.
- 1990—Additional stripping of overburden was completed at the B and WD Vein Complex, detailed (5-m spaced) sampling of the B vein was completed, and 58 HQ-sized diamond drill holes (5,158 m) on 50 m centres along the vein were drilled. Several gold shoots and locally stacked, parallel veins were identified. In the Siwash Lake area trenching, overburden stripping and drilling of four HQ-sized diamond drill holes (259 m) were completed. Soil sampling was completed in other areas of the Project, and geophysical surveys were done on the Agur Option.
- 1991—37 HQ-sized diamond-drill holes (6,587 m) were drilled in the B and WD Vein Complex, trenching was completed in the End Zone (200 m southwest of Siwash Lake), and aerial photos were taken.
- 1992—A 2,040 tonne bulk sample grading 137 g/t Au was excavated from the “Mother Shoot” (Main Vein in B Vein Complex), and 79 reverse circulation (RC) drill holes (2,683 m) were drilled to test the area of the open pit bulk sample.
- 1993—Open pit bulk sampling continued, 11 RC drill holes (942 m) were drilled in the pit area, a portal was collared, and a decline (480 m) driven for underground exploration.
- 1994—Underground exploration continued with 84 NQ-sized diamond drill holes (5,011 m) drilled underground. The main decline was extended 330 m with a branch of it extended an additional 185 m. Testing of underground mining methods (longhole stoping and shrinkage stoping) were pursued, and following receipt of a mining permit, the open pit was expanded.
- 1995—217 NQ-sized diamond drill holes (7,612 m) were drilled underground from the decline, and a total of 98 HQ-sized diamond drill holes (6,289 m) were drilled from surface in the Property, 70 of which (4,645 m) were drilled in the Siwash North area.
- 1996—88 NQ-sized diamond drill holes were drilled (6,946 m), five of these holes (1,120 m) were drilled in Siwash North Deep B, and 38 of the holes (1,997 m) were drilled to the east of the open pit area.
- 1997—Mining reclamation and site clean-up were the main activities at site.
- 1997–1999—Limited prospecting, sampling, and environmental monitoring.

- 2000—12 NQ-sized diamond drill holes (1,414 m) were drilled to test the WD, B, and Gold Creek veins.
- 2001—Six trenches (202 m) were completed in the Siwash East area.
- 2002—26 NQ-sized diamond drill holes (4,996 m) were completed in Waste Dump (WD) (7 holes), Deep B (11), Gold Creek West (GCW) (4), and Bullion Creek (4) zones.
- 2003—30 NQ-sized diamond drill holes (6,570 m) were completed, which tested strike and down-dip extensions of the WD zone; and claim posts in the south destroyed by logging were located with GPS.
- 2004—500 m of roads were built, 40 m of road-cut was mapped and sampled as a trench, 15.8 km of ground magnetometer geophysical survey was carried out to the east of the mine site and existing drill grid, additional claim posts were surveyed by differential GPS, and 44 NQ-sized diamond drill holes (10,265 m) were drilled in the WD, B, and BC areas.
- 2005—36 NQ-sized diamond drill holes (8,395 m) were completed in the WD, B, and Siwash Lake zones. A 6 m x 710 m (0.42 ha) or 350 m access road was cut to the proposed Bullion Creek site. The drill pattern in WD Vein was brought to 25 m by 50 m and the vein system was tested down-dip and to the west.
- 2006—58 NQ-sized diamond drill holes (8,873 m) were drilled to test the WD (27 holes), B (17 holes), and Siwash East (4 holes) zones. Trenching and soil sampling were done on new claims to the northwest side of the Property.
- 2007—9 HQ-sized diamond drill holes (2,470 m) were drilled on the A, B, C, and the deeper WD veins for metallurgical test samples. Metallurgical testwork was completed by G&T (2008). Construction of a 150 m temporary access.
- 2008—No exploration activities occurred on the Property.
- 2009—Pole-dipole IP and magnetometer surveys were conducted on lines spaced at 400 m intervals.
- 2010—A diamond drill program in November consisted of 87 drill holes in the resource area, for a total of 12,749 m. The program was primarily to provide infill drill holes to increase the drill-hole density in the areas of Inferred Mineral Resources. Additional pole-dipole IP was conducted to expand the 400 m grid surveys. Aerial photography and LiDAR topographic mapping were conducted by Eagle Mapping to create an orthophoto and 1 m contour map. Control for this survey and 2010 drill-hole collar locations were surveyed by a BC-registered land surveyor.
- 2011—GMMC completed a 77-hole 14,000 m diamond drill program that was designed to test the limits of the Siwash resource area along strike and down dip, as well as further drilling on the Gold Creek, Lake, and South zones. Drill collars from all years that could be found were surveyed using survey grade RTK GPS instruments capable of centimetre accuracies. A Titan 24 IP geophysical survey was completed, and pole-dipole IP surveying continued.

- 2012—GMMC completed 13,920 m in 123 diamond drill holes on the Siwash, Nicola, Bullion, Lake, End, Discovery, and South Zones. The pole-dipole IP geophysical surveying was completed, and a bulk sample test mining operation was started. That year open pit mining removed 38,582 m^3 of soil, 53,007 m^3 of non-mineralized waste rock, and 329.5 m^3 of mineralized vein material.
- 2013—GMMC completed 4 holes for 1,678 m of percussion drilling in South, Lake, and Siwash North Zones. Chip logging established the holes' general geology, and assays confirmed the presence of gold down-dip of core drill intersections. Geological mapping and prospecting occurred regionally on the Property. A test shipment of 504 tonnes of vein material with an average grade of 11.97 g/t Au and 35 tonnes of historical stockpiled material with an average grade of 14.66 g/t Au was processed at the Kettle River Mill in Republic, Washington.
- 2014—The bulk sample test-mining operation continued. A shipment of 6,092 tonnes of vein material with an average grade of 17.03 g/t Au and 1,130 tonnes of historical stockpiled material with an average grade of 4.09 g/t Au was processed at the Kettle River Mill. All rock storage piles and berms were re-contoured, and 0.5 m of soil spread.
- 2015—No exploration occurred on the Property.
- 2016—No exploration occurred on the Property.
- 2017—No exploration occurred on the Property.
- 2018—No exploration occurred on the Property.
- 2019—A high-resolution helicopter-borne airborne magnetometer and radiometric survey was flown over the Property's central section. The survey coverage amounts to ~1,108-line km spread over 7,233.875 ha.

6.2 Mining History

In the early 1990s, Fairfield initiated open pit and underground mining in the Siwash North zone. The following history of that mining is drawn from Jakubowski (2006). In 1992 a 2,040-tonne bulk sample grading 137 g/t Au was excavated from the "Mother Shoot" (Main Vein in B Vein Complex). This bulk sample was shipped to the Horne Smelter in Rouyn-Noranda, Quebec, for smelting.

In 1993, open pit mining continued, and a portal was collared. A 3,387-tonne bulk sample grading 105.6 g/t Au was extracted from the open pit and shipped in three batches to Asarco's smelter in Helena, Montana. A 480 m decline at −15% was driven that year for underground exploration, and two vein drifts were developed for test mining. The drift on the 1570 level produced approximately 140 tonnes grading 38 g/t Au before the drift was abandoned and refilled due to poor ground conditions. Three raises at 5 m centres, for 36 m, were driven off the 1611 level drift. The pillars were subsequently stoped to produce about 315 tonnes of mineralized quartz vein grading 70 g/t Au.

In 1994, Fairfield received a mining permit and expanded the open pit, extracting 9,180 tonnes of vein material grading 91.5 g/t Au. Underground, the 1611 drift was extended westward, five additional

raises added, and the initial three raises lengthened up to the 1620 level. About 1,200 tonnes of quartz vein material grading 78 g/t Au were extracted. The main decline was extended 330 m after the open pit was completed, with a branch decline extended an additional 185 m. Testing of underground mining methods was pursued (longhole stoping on the 1584 level and shrinkage stoping on the 1589 level). The longhole stope produced about 95 tonnes grading 16.5 g/t Au from the drifting, but due to the dilution most of this material was left in the stope. The shrinkage stope was about 30 m of drift by 6 m of raise. Approximately 105 tonnes at 15 g/t Au was hauled to the surface but the bulk of the mined material remains in the stope. A single shipment of 1,800 tonnes of mined material was sent to the Asarco mill, while 2,300 tonnes remained on site at surface.

In 1995, all wiring and piping was removed from the underground workings and the portal was sealed. No further mining occurred until the recent work by GMMC.

In early 2012, GMMC applied to the BC Ministry of Energy, Mines & Petroleum Resources to reactivate *Mines Act* Permit M-199, and in late August the permit was amended to allow collection of a 4,000 m^3 bulk sample. The major focus for the bulk sample was to confirm the results of the PEA (Pooley et al., 2011). The test mining and bulk sample program were also used to refine the mining technique and confirm parameters for full-scale operations. The efforts in 2012 were to remove most of the hanging wall non-mineralized rock, and in doing so a small amount of mineralized vein material was recovered. This new mining is from an open pit referred to as Pit 2 and is east of the historical pit mined in the 1990s (now referred to as Pit 1). In total, 2012 open pit mining from Pit 2 removed 38,582 m^3 of soil, 53,007 m^3 of non-mineralized waste rock, and 186.8 m^3 of mineralized vein material.

Although no mining occurred in 2013, the first lot of mined vein material was processed at a toll mill owned and operated by Kinross's subsidiary Echo Bay Mines Company. A 500-tonne test sample was shipped to and processed at the Kettle River Mill.

Results of milling Lot 1 were encouraging, and test mining operations resumed in 2014 within Pit 2. That year, 2,256.5 m^3 of mineralized vein material were mined, and a shipment of 6,092.5 tonnes of vein material with an average grade of 17.03 g/t Au and 1,130 tonnes of historical stockpiled material with an average grade of 4.09 g/t Au was also processed at the Kettle River Mill. A summary of all mining from 1992 to 2014 is presented in Table 6-4, including tonnes processed, average grade, and ounces recovered.

It should be noted, and the reader cautioned, that the mineralized quartz vein material mined in the 1990s was much higher grade than that mined in 2012 and 2014. East of Pit 1 the vein splits into multiple branches which are thinner and lower grade than the "Mother Shoot." This thinner and lower grade "B, or 1300" vein is what was mined in Pit 2 during 2012 and 2014. Higher-grade and wider zones within the 1300 vein still exist below Pit 1.

Table 6-4: Gold Production Summary

Date	Mining Method	Tonnes Processed	Average Grade (g/t)	Ounces Recovered
1992	Open Pit	2,041	137	9,000
1993	Open Pit	3,388	106	11,500
	Underground	408	61	800
1994	Open Pit	9,181	91	27,000
	Underground	1,170	78	2,950
1995	Landing Cleanup	113	27	100
	Underground	200	17	110
2012	Historical Stockpile	35	14.66	17
2014	Historical Stockpile	1,130	4.09	149
2012	Open Pit	504	11.97	194
2014	Open Pit	6,092	17.03	3,337
	Historical Open Pit Total	**15,889**	**93.50**	**47,765**
	Historical Underground Total	**1,778**	**67.52**	**3,860**
	Bulk Sample Open Pit Total	**6,597**	**16.65**	**3,531**
	Total	**24,263**	**70.71**	**55,156**

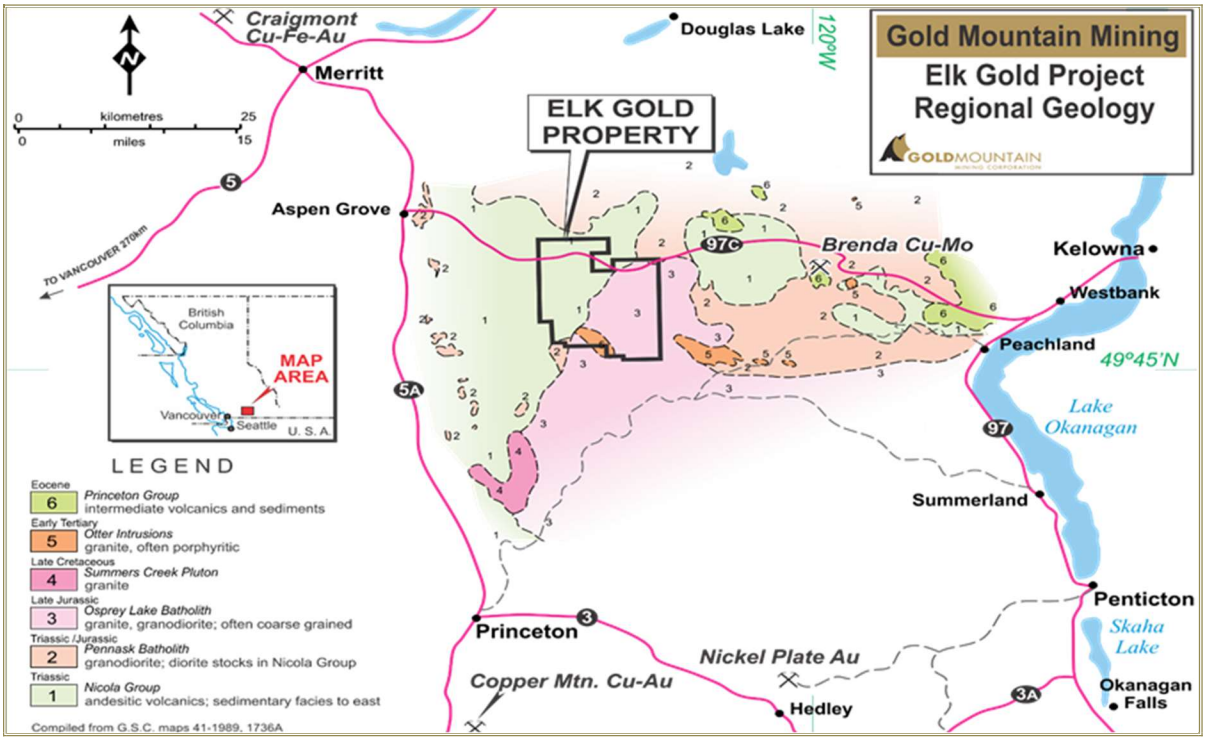
7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting

The Elk Gold Project is situated in the Intermontane (tectonic) Belt of south-central BC. Rice (1947) of the Geological Survey of Canada (GSC) completed regional mapping. Monger (1989), also with the GSC, compiled the Hope (92H) map sheet geology at 1:250,000 scale. Recent mapping by Mihalynuk et al. (2015) of the BC Geological Survey Branch (GSB), as part of the Southern Nicola Arc Project (SNAP), re-examined the Nicola group rocks previously examined by Preto (1979), also with the GSB. While this latter mapping included the Property, the SNAP fieldwork was focused on Nicola Group volcanic rocks, and only a cursory review was made of Property area intrusive rocks on the far east side of their map area.

Rice (1947) and Monger's (1989) maps depict the Property to be underlain by Triassic-age Nicola Group volcanic–sedimentary rocks in the western third of the Property; Jurassic-age granitic rocks of the Osprey Lake Batholith underly the eastern two-thirds of the Property. Feldspar-porphyry stocks and dykes of the Upper Cretaceous Otter Intrusions occur in the southwest claim area and cut both Nicola Group volcanic rocks and Osprey Lake granitic rocks. Tertiary andesite dykes intrude all the above. Gold appears to be spatially related to the andesite dykes and contained within pyritic quartz veins which locally cut the dykes. Figure 7-1 is after Monger's map compilation.

The Nicola Group as described by Preto (1979) consists mainly of mafic flows, pyroclastic rocks, volcanic breccias, epiclastic rocks, and, locally, argillite and limestone. The volcanic rocks are quartz-saturated (but rarely quartz-bearing), clino-pyroxene (±plagioclase) porphyritic basalts, locally with analcime. The Nicola Group has been divided into four lithological belts by Monger (1989, 1992) and Monger and McMillan (1989). These include: 1) a western belt of steeply dipping, east-younging, late Carnian to Norian, subaqueous felsic, intermediate, and mafic calc-alkaline flows grading up into volcaniclastic rocks; 2) a central belt of early to middle Norian, subaqueous to subaerial basalt and andesite flows, volcanic breccias, and laharic breccias of both alkalic and calc-alkalic affinity; 3) a younger, westerly dipping, "eastern volcanic belt" (Late Norian) composed of subaqueous and subaerial alkali, intermediate and mafic flows, volcanic breccias, and epiclastic rocks that were deposited on or between emergent volcanic edifices; and 4) an eastern sedimentary assemblage (Ladinian to middle Norian) that is overlapped by the eastern volcanic belt, and consists mainly of greywacke, siltstone, argillite, alkalic intermediate tuff, and reefal limestone, which may record a back-arc basin.



Source: Bayshore, August 2020.

Figure 7-1: Elk Gold Project Regional Geology

7.2 Property Geology

7.2.1 Lithology

Nicola Group volcanic rocks mapped on the Property comprise massive and locally porphyritic basalt and andesite; pale grey-green, siliceous layered tuff and green to pale-green agglomerates. In places the volcanic rocks are silicified and subject to carbonate or epidote alteration. Amygdaloidal textures with carbonate-filled voids have been observed in outcrop and drill core. Disseminated pyrite within the Nicola Group is common.

Osprey Lake batholithic rocks vary from granite to quartz monzonite to granodiorite in composition and comprise the bulk of the intrusive rock on the Property. Granodiorite is predominantly found on the margin of the batholith and has been historically described as a granodiorite to diorite chill margin. The quartz monzonite is pinkish-grey in colour, medium- to coarse-grained, and equigranular to locally porphyritic. Aplite and andesite dykes intrude the batholith. Osprey Lake rocks have also been observed intruding Nicola Group rocks.

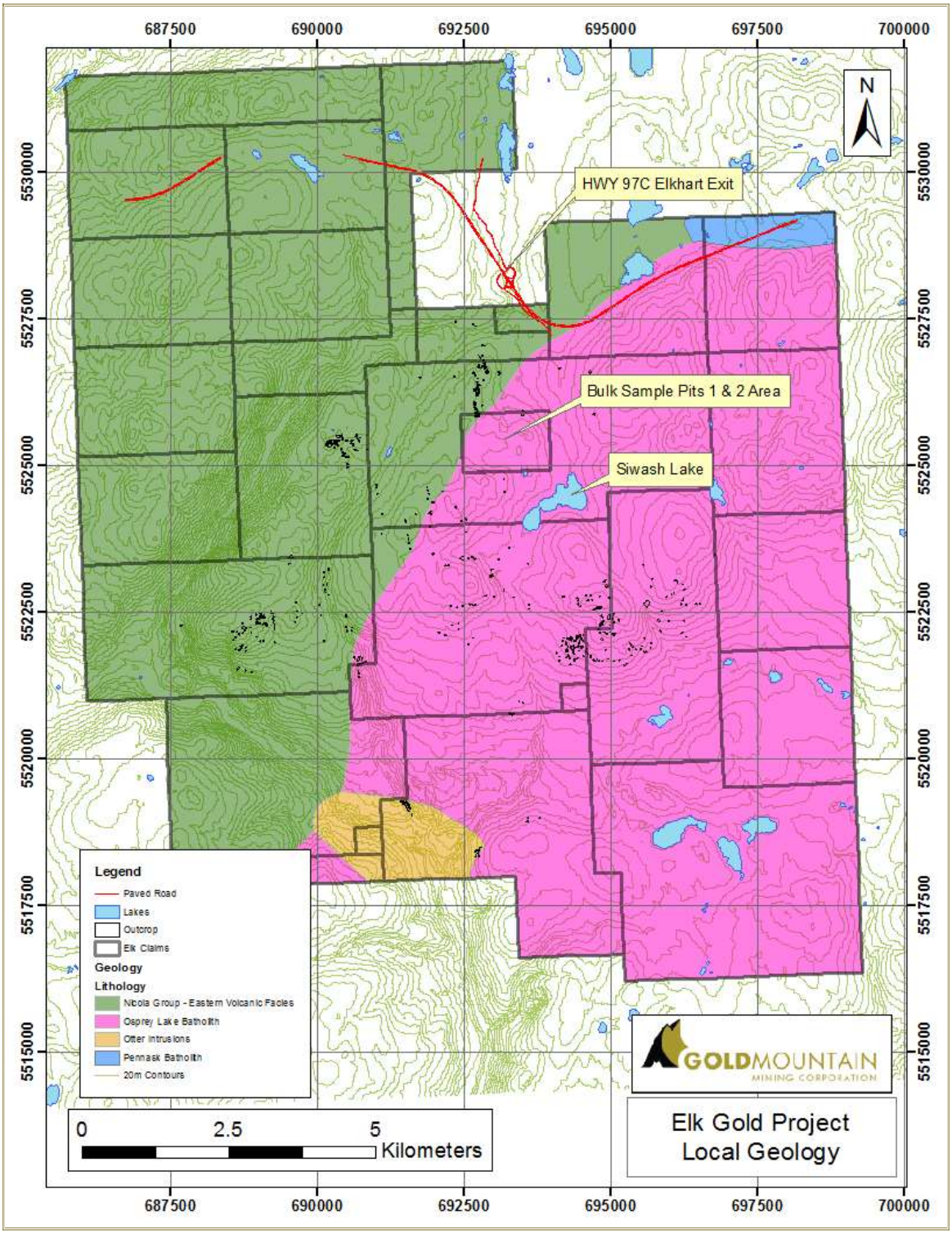
The Otter Intrusions consist of quartz-feldspar porphyry, feldspar porphyry, and quartz-biotite-feldspar porphyry dykes and stocks. Locally the quartz-feldspar porphyry is extensively clay altered. Otter Formation rocks, which are younger than, and intrusive into Osprey Lake rocks, have been

mapped near the southern Property boundary. Boundaries of these rocks in the southern part of the Property were revised after 2013 surface mapping (Figure 7-2).

Andesite dykes are dark grey-green, fine-grained, and vary in thickness from 30 cm to 5 m. They are thought to be associated with a later phase of the Otter Intrusions (Rowe, 1995) and have been intersected by drill holes throughout the Property.

Gold mineralization occurs within quartz-sulphide veins and stringers, most often within phyllic- and silica-altered Osprey Lake rocks and adjacent phyllic- and silica-altered Nicola volcanic rocks. Pyrite is the most common sulphide (Conroy, 1994), ranging from 5% to 80%, with higher percentages often associated with chalcopyrite and tetrahedrite. Gold occurs as fine-grained free gold (typically less than 50 µm) in quartz, and also within quartz-pyrite boxwork, and in fractures within veins (King, 2001). Gangue minerals include quartz and altered wall rock clasts (xenoliths), with minor amounts of ankerite, calcite, barite, and fluorite. Most of the previous mine production occurred within the quartz-monzonite and granodiorite border phase of the batholith (Lewis, 2000).

The Property contains nine named zones with gold mineralization. Those that have had the most extensive drill testing are described in the following sections. In general, the lithology, mineralization, alteration, and structure are common to each and are summarized within the Siwash North zone description. This PEA focuses on the Siwash North zone and includes a resource calculation, so individual results have been omitted within this section.



Source: Bayshore, August 2020

Figure 7-2: Property Geology

7.2.2 Siwash North Occurrence

The Siwash North 1000 and 2000 vein complexes (historically called the B and WD Siwash Veins) are emplaced within fault/fracture zones that strike east–northeast and dip moderately to steeply southward (Jakubowski, 2006). The veins are cut by a north–northwest striking, near vertically dipping mafic dyke known as the RBF. The dyke was exposed during 2012 bulk sample mining and is a 3 m wide faulted and altered post-mineral structure that divides mineralized veins into two regions with little apparent offset.

In the Siwash North zone, Pit 1 (which is west of the RBF), the 1300 vein is a single, moderately-dipping high-grade gold vein. In Pit 2, which is mostly east of the RBF, the 1300 vein occurs as multiple vein and gouge clusters which host lower-grade concentrations of gold. While most mined veins in Pit 2 were east of the RBF, the RBF was exposed while establishing the main access ramp, and single vein material was removed west of the RBF.

7.2.2.1 Structural Geology

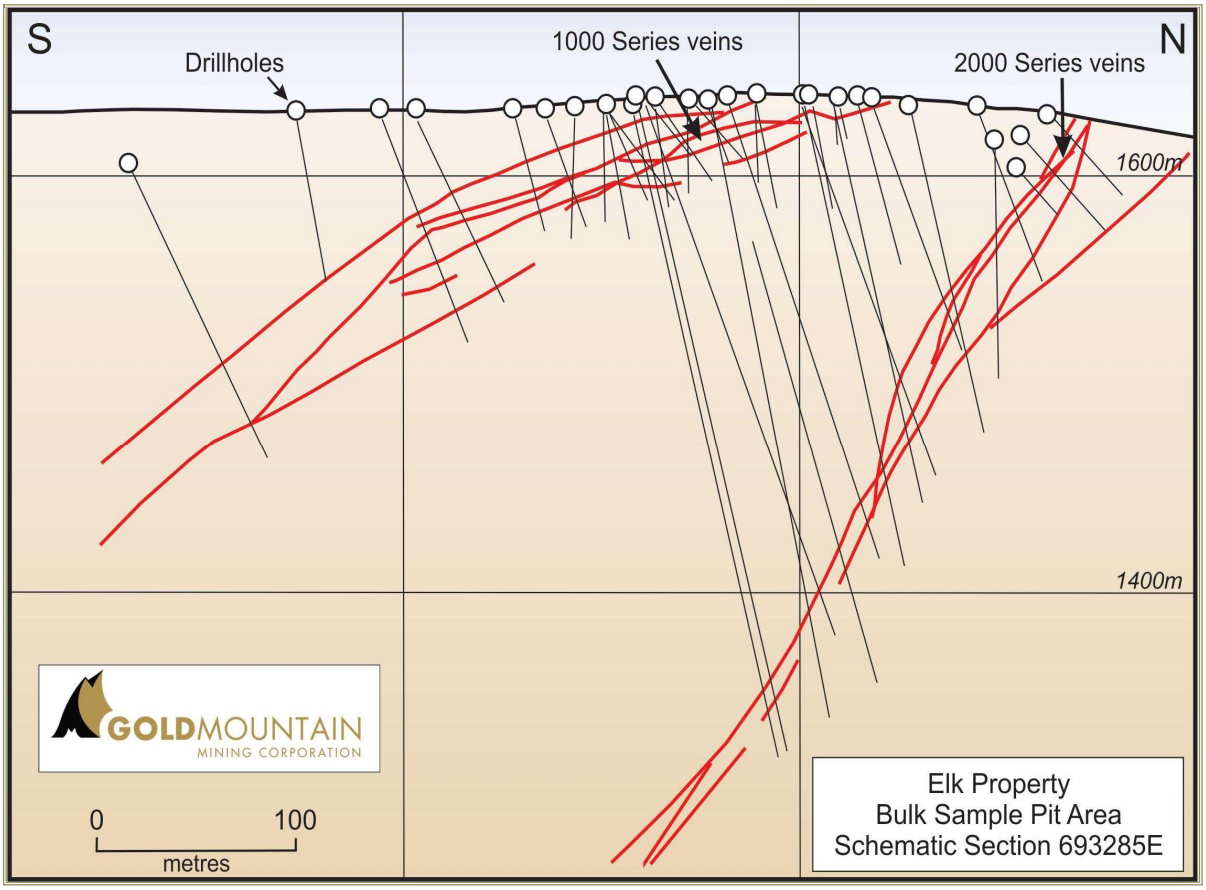
The Nicola Group on a regional scale dips 60° west, forming the east limb of a syncline that trends north–south with its axis nearly 5 km west of the Property (Rice, 1947). On the Property, the contact between the Nicola volcanic rocks and the Osprey Lake granodiorite trends northeast, and dips moderately to the east.

The Siwash North gold-bearing quartz-sulphide veins strike east–northeast and dip moderately to steeply south. Veins cut shallowly across the intrusive contact between Osprey Lake Batholith and Nicola Group volcanic rocks, but the vein is sporadic in nature within Nicola rocks.

A fracture-fault system trending 060° to 080° and dipping to the southeast at 20° to 75° controls the Siwash North vein system (Jakubowski, 2006). A secondary fracture set, trending 110° to 160° and dipping 45° to 60° southwest, hosts the Gold Creek West Zone. Gold Creek West is located 400 m southwest of the Siwash North vein system. It has been postulated that the Siwash North B Vein Complex, renamed 1000 series, and the WD Vein Complex, renamed 2000 series (Pooley et al., 2011), could be part of an anastomosing vein system that could have similarities in geometry to a Riedel shear system (Alexander, 2011). Figure 7-3 is a schematic diagram showing the relationship of the 1000 and 2000 series veins.

The exposure of veins in Pit 1 show that they vary in structural style and may be divided into the west, central, and eastern segments (Lewis, 2000). As further described by Lewis, the western segment includes a system of sub-parallel en-echelon veins indicating oblique, sinistral, and minor normal movement. The central segment is a single structure—the main target of mining in the early 1990s—which appears to transition from the en-echelon veins in the west to a series of sub-parallel splays to the east. In general, the near-surface dip of these veins is shallow, but steepens with depth. The true nature of this dip change (known either as the rollover zone or the hinge) is unknown despite the

underground work. It has yet to be shown whether this is due to intersecting faults, or tectonic bending.



Source: Bayshore, August 2020

Figure 7-3: Schematic Section 693285E Showing Postulated Vein Relationships

7.2.2.2 Alteration

This section is taken from Benoit, Wilson, and Daubeny (2012). Nicola Group volcanic rocks are locally silicified, with associated sericitic alteration, and subject to patchy to pervasive carbonate and epidote alteration (Figure 7-4). This propylitic alteration can also consist of calcite, chlorite ± epidote ± albite and occur in varying intensity throughout the Nicola Volcanic rock section (Benoit, Wilson, & Daubeny, 2012). In 1989, a drilled sample submitted for petrographic study was characterized as metamorphosed andesite dominated by plagioclase, with little hornblende, and containing replacement patches of K-feldspar with much less plagioclase and hornblende (Payne, 1990). A second sample of a strongly altered, slightly porphyritic andesite dyke contained plagioclase phenocrysts replaced by sericite-quartz, with a groundmass of moderately abundant secondary K-feldspar.



Figure 7-4: Altered Nicola Group Volcanic Rocks

Plutonic rocks from the Osprey Lake Batholith are pinkish grey, medium- to coarse-grained equigranular quartz monzonite with a granodiorite border phase adjacent to Nicola volcanic rocks. Osprey Lake rocks show weak to strong phyllic alteration, with plagioclase altered to sericite (and locally ankerite), and biotite mainly replaced by pseudomorphic muscovite with Ti-oxide lenses along cleavage (Payne, 1990). In more-strongly altered rocks the stable assemblage is quartz-muscovite-pyrite. Plagioclase and K-feldspar are altered completely to muscovite/sericite with patches of ankerite. Biotite is altered to pseudomorphic muscovite in part, with patches of sericite, ankerite, and quartz. Hornblende is altered completely, mainly to carbonate and chlorite or carbonate and sericite.

Where quartz veins are exposed by trenching or mining, they are accompanied by a number of different alteration assemblages, including phyllic, sericitic, and silicic. Propylitic and argillic alteration can also occur near veining as well, but these do not appear to be directly associated with mineralization. Quartz monzonite alteration envelopes outboard of any clay selvage material are initially silica dominated, with silica giving way to sericite and then salmon-pink secondary K-spar, typically over intervals of a few centimetres or tens of centimetres (Figure 7-5) (Daubeny, 2012). The observed K-spar is likely the microveinlets described in the 2010 petrographic report (Leitch, 2010). During core logging, it appeared to be a secondary feature, but under thin section it was seen as a primary feature within the intrusion.

Another petrographic sample from within the Osprey Lake intrusive complex suggests that potassic alteration is not present in this batholith, but rather that areas appearing to have secondary K-feldspar display a poikilitic texture with the K-feldspar occurring as the oikocrysts. This is to say that larger K-spar crystals surround all other crystals, or just simply fill the cracks between the other crystals present, and thus the K-feldspar present is primary (Leitch, 2010).



Figure 7-5: Vein Alteration Envelopes within Quartz Monzonite Host Rocks

Strong phyllic, sericitic, and silicic alteration is generally observed marginal to the veins and associated gold mineralization. Alteration envelopes within diorite are similar to those of the quartz monzonite; however, the most common diorite-hosted alteration may comprise a continuum whose end members are hairline Fe-carbonate veinlets accompanied by millimetre-scale Fe-carbonate envelopes or sericite veinlets accompanied by K-spar-dominated envelopes; with the Fe-carbonate alteration type being the most prevalent (Daubeny, 2012).

Not all alteration assemblages are always present. Propylitic and argillic alteration may be spatially associated with mineralization, but not genetically. There are several drill holes where propylitic alteration is ubiquitous and does not seem to be associated with mineralization at all. Alteration descriptions are provided in Table 7-1.

Table 7-1: Alteration Descriptions

Alteration	Description
Propylitic	Light green, with biotite and hornblende altered to chlorite and plagioclase saussuritized. In volcanic rocks the colour is generally olive green, and rocks are soft
Argillic	Bleached rock with plagioclase white and clay altered, and potassium feldspar slightly altered. Volcanic rocks are bleached to light green or grey
Advanced Argillic (Fault Gouge [?])	Often sheared and white in colour with most or all feldspars destroyed, and quartz is free floating. Often associated with quartz veining and may be associated with structural movement. Volcanic rocks are white or blue in colour
Sericitic	Pale green with a micaceous sheen, with plagioclase altered to sericite. This style of alteration is often associated with quartz veins and appears to be the lowest-grade alteration associated with gold mineralization
Potassium Feldspar Stable Phyllic	Light pink-green or yellowish with potassium feldspar fresh, pink, and blocky. Plagioclase and mafic minerals are altered to fine-grained quartz-sericite-pyrite. This style of alteration is often associated with gold mineralization in veins.
Phyllic	Generally grey, fine-grained quartz-sericite-pyrite alteration usually associated with gold-bearing quartz veins.
Silicic	Quartz veining or replacement with moderate conchoidal fractures

7.2.2.3 Mineralization

Mineralization is thought to be Tertiary in age and associated with the intrusion of the Otter dikes and stocks (Jakubowski, 2006). Gold occurs primarily in native form, in places as aggregates of fine flakes in quartz, in quartz-pyrite boxwork and in fractures within the vein. Veins are dominated by quartz, with patches of sulphides, muscovite, and ankerite (Payne, 1990). Sulphides are dominated by pyrite, with traces of pyrrhotite/marcasite and arsenopyrite as seen in polished thin sections. Traces of chalcopyrite, sphalerite, and galena are seen in quartz veins, rarely ranging up to percent grades in some samples.

Native gold occurs in some quartz-sulphide veins, mainly in fractures in pyrite grains (Payne, 1990). In the 1989 petrographic samples, most native gold grains were not associated with base metal sulphides. In a few samples, gold is intergrown coarsely with galena (in blebs in pyrite) and with galena-tetrahedrite (in a fracture in pyrite). In general, there seems to be a correlation between bismuth, copper, zinc, and gold. Native gold grains are mainly from 0.03 mm to 0.10 mm in size, though grains up to millimetre size have been photographed.

At surface, most of the sulphides are leached out, leaving only minor pyrite and chalcopyrite in quartz (Jakubowski, 2006). Gold is closely associated with pyrite and a blue mineral (possibly a gold-bismuth alloy called maldonite) or a copper-bismuth-antimony sulphosalt. Metallic minerals in the quartz veins (listed in decreasing abundance) are pyrite, chalcopyrite, sphalerite, galena, tetrahedrite, maldonite(?), pyrrhotite, and gold (Jakubowski, 2006). Gangue minerals consist primarily of quartz and altered wall-rock clasts. Ankerite and lesser amounts of calcite, minor barite, and fluorite occur locally.

Quartz veins on the Property measure from less than 1 cm to (rarely) over a metre in thickness, with 8 cm to 15 cm being the most common width observed in drill core. Veins vary in composition from white, fluid-inclusion-rich and barren of sulphide to being crudely banded smoky-grey and sulphide-bearing. Vein-hosted sulphide minerals consist dominantly of pyrite, often in concentrations of 5% to 10% and rarely to more than 50%, but can also include occasional traces to 5% chalcopyrite, sphalerite, and minor galena. Quartz veins can consist of pure quartz ± sulphide material or contain varying amounts of wall rock as inclusions that, if present, are usually green, sericite-altered, and silicified to the point of textural destruction. These veins occur in all of the rock units mapped on the Property (including some early dykes), but are more prevalent and best developed in the granodiorite and quartz monzonite that encompasses the historical and current bulk sample open pits.

7.2.3 South Showing

The host rock for mineralization is Osprey Lake quartz monzonite. The central and western sections of the South Showing contain four distinct veins identified by drilling and trenching. Typically, two or three of these veins are seen in each drill hole, but there are holes in which all four are seen. In general, there are two zones of mineralization spaced 20 m to 40 m apart, each of which contains one distinct quartz vein or a cluster of small quartz veins carrying gold mineralization, as exemplified in drill hole SND11-131. Within SND11-131 the first two gold-bearing vein structures are about 4.5 m apart, and the third and fourth vein structures are approximately 30 m down-hole of the first two, and are more closely spaced with higher gold grades over approximately 0.9 m.

Heavy silica alteration and silica flooding of the host quartz monzonite is sometimes found proximal to mineralization; however, it is unclear how this is related to the silica-healed breccia found on the east end of the zone. Higher gold grades in quartz veins are more frequently associated with areas of strong silica alteration and flooding than within veins with no alteration envelope, suggestive of a causal link. The breccia found on the east end of drilling has been seen in several 2011 and 2012 drill holes with varying widths and grades. It can be described as a wide, low-grade (i.e., <1 g/t Au) mineralized silica breccia that includes higher-grade gold zones within. The breccia is dominantly quartz monzonite fragments and minor mafic dyke fragments cemented in pyrite-mineralized silica. The breccia appears irregular in shape and may represent a pipe created at an intersection of faults, fractures, dykes, or other structural breaks in the host quartz monzonite, though its geometry remains unknown.

Trenching (14 in total) has exposed narrow quartz veins, variably altered andesite dykes, and a breccia hosted in quartz monzonite of the Osprey Lake formation over a strike length of 600 m. Several chip sampling programs returned values up to 16.6 g/t Au over 45 cm, up to 7.92 g/t Au over 1.0 m, and series of nine chip-and-panel sample strings taken from a stripped area over a strike length of 25 m returned an average of 3.81 g/t Au over 2.0 m.

The zone has been tested by 43 drill holes between 1995 and 2012. Eight holes drilled in 1995 (481 m) were completed on three sections (Azimuth 330°) spaced 75 m apart, which intersected pyritic quartz

veining associated with a breccia-cutting quartz monzonite. Assay results were variable over narrow widths and ranged from 3.81 g/t Au to 32.12 g/t Au. Drill holes SSD95-166 and -167 were drilled under Trench SS87-2 and returned variable results from sampled quartz veins, ranging from 1.13 g/t Au to 15.46 g/t Au.

Two holes drilled in 2010 returned results of approximately 1 g/t Au over composite widths of 3.2 m to 7.8 m.

Eleven holes were drilled in 2011, and gold mineralization was encountered both along strike and down dip of that previously known; in addition, a new mineralized zone was encountered and described as a quartz breccia. The quartz breccia is observed to be over 50 m wide in one hole and contains grades up to 31.7 g/t Au and 73.8 g/t Ag across 0.25 m in hole SND11-144. It is characterized by silica flooding with pyritic mineralization occurring mainly along breccia selvages where the host quartz monzonite has been included in the breccia. The strongest pyrite–gold mineralized areas seem to correspond with the on-strike extent from discrete veins in the area. This observation suggests that the pyrite and gold mineralization in both the veining and silica-flooded breccia originates from the same source, and conversely that the breccia and silica flooding were post-veining.

Twenty holes were drilled in 2012 to identify the extents of the quartz monzonite-hosted quartz breccia, as well as the gold-bearing veining system local to the area. Eleven of these holes were drilled within the area where the dominant lithology is quartz monzonite. Pervasive silica flooding and sericitic alteration appear to be controlling factors on high gold values in drill samples. The breccia orientation is estimated to have a strike of 060° and dip of 40° toward the south. Hole SSD12-226 intersected a 2.5 cm quartz vein within the quartz breccia system carrying 30% pyrite and 5% chalcopyrite that graded 230 g/t Au over the 0.5 m sample width, and in combination with other lower-grade sample sections yielded a composite grade of 8.01 g/t Au and 14.38 g/t Ag over 15.75 m. Four additional holes have assayed gold grades greater than 1 g/t over widths greater than 4 m within the silica-healed breccia, as follows: SSD12-230—2.81 g/t Au over 6.35 m; SSD12-228—2.37 g/t Au over 4.87 m; SSD12-232—2.38 g/t Au over 5.27 m; and, SSD 12-218—2.42 g/t Au over 4.27 m.

Nine holes were drilled to the southwest of the 2011 drill holes. No breccia was encountered in any of the nine drill holes. Hole SSD12-244 encountered a fault at 121.7 m, which may partially explain the reason for the lack of the quartz breccia to the southwest of the zone. Veining was the most prevalent form of mineralization on the southwest section, yet silica flooding was only encountered once, in Hole SSD12-260. Widths and grades vary from narrow high-grade zones (SSD12-246 with 55.3 g/t Au over 0.85 m) to wider zones with lower grades (SSD12-260 with 1.39 g/t Au over 9 m).

On the basis of surface mapping and sampling and interpretation of drill-hole data, it is still unclear as to what the geometry of the breccia zone is, but current information seems to fit into pipe-like geometry that dips to the south around 75°. It appears that the veining in the area continues through the breccia zone and that the highest grades correspond with locations in the breccia where the veins

were expected, which may suggest that the breccia event was pre-mineralization, or syn-mineralization.

The South Zone remains the second-largest area covered by exploration, measuring approximately 400 m of strike length and up to 150 m of dip length, and is open along strike and down dip on all sides.

7.2.4 Lake Zone

The Lake Zone is located 800 m south of Siwash North. Gold mineralization occurs in two or three distinct zones containing one to three quartz stringers, and veins ranging in size from <1 cm up to 35 cm wide, within strongly argillic- and phyllic-altered quartz monzonite of the Osprey Lake Batholith. Although there are many mineralized zones within this area, one typically located within 5 m of a mafic dyke is the main target and contains the highest grades. The mineralized zone strikes east–west for almost 400 m, dips south at about 50°, and is closely associated with an andesite dyke. The mineralized zone is open along strike and down dip. Gold is mainly associated with pyrite and chalcopyrite, although local galena and sphalerite can be found up to 5% along with minor (i.e., <1%) tetrahedrite and maldonite(?). Silver values are much higher than at Siwash North, probably due to the higher galena content.

The Lake Zone area contains numerous gold-in-soil anomalies which range up to 1,210 ppb Au. Two of these anomalies were tested in trenches SL89-1 and SL89-2, in which argillically altered pyritic andesite dyke returned values up to 12.69 g/t Au over 0.86 m. In 1990 the exposure was increased by 17 m along strike. Panel sampling in SL90-1 returned values up to 30.69 g/t Au over 0.34 m. Also, in 1990, Trench SL89-2, located 150 m west of SL89-1, was extended 100 m to the south (SL90-2) and exposed the mineralized zone from which a grab sample yielded 40.39 g/t Au.

In 1995, seven drill holes totalling 477 m were drilled on three north–south sections spaced 50 m and 100 m apart. The vein, which is hosted in quartz monzonite adjacent to an andesite dyke, was intersected in all holes and returned values up to 55.44 g/t over 0.55 m.

In 2011, eight drill holes (1,125 m) were drilled in this area to test for possible down-dip and strike extensions. A pyritic quartz vein was intersected in SND11-150 which assayed 4.43 g/t Au over a 1.2 m sample width including 26.6 g/t Au and 61.9 g/t Ag over a 20 cm width. Drill hole SND11-151 pierced the same vein as SND11-150 and encountered another 1.2 m interval with 5.12 g/t Au, including a higher-grade 20 cm vein assaying 30.7 g/t Au and 16.5 g/t Ag. These intersections are thought to be dip extensions of each other and were encountered immediately above a texture described as vuggy, boxwork pyrite. It is inferred that this vein occurrence is the same high-grade vein seen in Trenches SL90-1 and SL90-2.

In 2012, five holes, spaced along sections 100 m apart, were drilled with a northerly azimuth at dips between −50° and −75°. Hole SLD12-209 intersected two quartz veins at 14.9 m and 17.9 m with a composited assay grade of 5.08 g/t Au and 1.64 g/t Ag over 4.66 m (including 70.4 g/t Au and 17.6 g/t Ag over 0.33 m). Drill hole SLD12-213, located 100 m south of SLD12-209, yielded a

composited 14.07 g/t Au over 3.95 m (including 89.0 g/t Au over 0.3 m) from a 19 cm thick quartz vein that hosted coarse-grained and banded pyrite as well as a trace chalcopyrite. The vein is cut by a dyke which is interpreted to be post mineral, and splits what now looks like two quartz veins; the veins include up to 6% pyrite and 0.5% chalcopyrite as well as an altered hanging wall.

In SLD12-215 at 102 m depth a sample was taken that returned a grade of 22.9 g/t Au and 66.5 g/t Ag across 0.5 m which included two veins, one being 2.5 cm wide and the other 0.5 cm. SLD12-274 was located 70 m east of SLD12-213 and 215 m and 130 m south of trench SL90-1. A vein system 47.5 m to 48.5 m down-hole assayed 1.35 g/t Au and another at 108 m graded 7.11 g/t Au over 0.5 m. The lower intersection corresponds with the veining seen in trench SL90-1.

Mineralization in the Lake Zone remains open both down dip and along strike.

7.2.5 Gold Creek Zone

The Gold Creek Zone is located approximately 150 m south of the west end of Siwash North and was initially investigated to follow up on soil geochemical and very-low-frequency electromagnetic (VLF-EM) geophysical anomalies. Mineralization is similar to Siwash North, in that gold-bearing pryitic quartz veins are hosted in the Osprey Lake quartz monzonite.

In 1996, four holes totalling 399 m tested gold-in-soil anomalies and a VLF conductor with assays up to 5.68 g/t over 0.82 m. Another seven holes totaling 556.9 m tested a gold-in-soil anomaly in the Gold Creek West area. A quartz vein intersected in the Gold Creek West area was defined by drilling over a strike length of 160 m with grades up to 19.58 g/t over 0.3 m.

Five holes drilled in 2000 returned values up to 16.55 g/t over 0.5 m. The vein steepens from a 30-degree dip to a 60-degree dip over 140 m along strike to the east.

The best results from four holes drilled in 2002 came from drill hole SND02-324 where a 0.62 m sample assayed 1.69 g/t Au.

In 2011, eight holes (906 m) were completed to test for the extension of mineralization down dip and along strike. These holes intersected some very-high grades over narrow widths, the most impressive of which was in SND11-155, which had a 20 cm interval assaying 84.6 g/t Au and 319 g/t Ag. Other grades were 78.5 g/t Au and 321 g/t Ag over 30 cm in SND11-153, 71.7 g/t Au and 153 g/t Ag over 25 cm in SND11-154, and 38.7 g/t Au and 100 g/t Ag over 20 cm in SND11-158.

The Gold Creek Zone remains open along strike and at depth.

8 DEPOSIT TYPE

The Elk Gold deposit is an intrusive-related, structurally controlled quartz vein system. The gold and silver-bearing quartz veins have historically been characterized as mesothermal, based on fluid inclusion studies. The fluid inclusions within quartz crystals in the veins indicate gold mineralization formed at a minimum temperature of 250°C and a pressure of 2.5 kilobars, corresponding to lithostatic pressure at a depth of 7 km (Geiger, 2000). Considerable open space within the veins not related to weathered pyrite suggests a much shallower vein emplacement and more study will be needed to resolve the apparent conflict. The vein systems consist of structurally controlled, narrow, pyritic quartz veins hosted in granitic as well as volcanic rocks near the contact between these two primary lithologies. Emplacement of quartz veins is thought related to Tertiary Otter intrusions. A possible sequence of events is as follows:

- Deposition of Nicola Volcanic Rocks
- Emplacement of Osprey Lake Batholith
- Emplacement of Otter Intrusions and related dykes
- Deformation caused fractures and fracture-fill precipitation of quartz to form veins with sulphides and associated gold mineralization causing vein envelope alteration
- Uplift and fracturing associated with orogeny, along with additional late stage alteration along structural weaknesses
- Erosion to present topography.

9 EXPLORATION

9.1 Historical Overview

Cordilleran was the exploration arm of Fairfield, and in 1986 staked the original claims on the present Elk Gold Project after discovering gold-in-quartz vein float on the edge of a recent logging landing. Prospecting, geological mapping, and soil sampling helped define trenching targets, and those trenches that intersected gold-bearing quartz veins became diamond-drill targets testing vein continuity to depth. Cordilleran conducted all exploration from 1986 to 1991 and Fairfield assumed operatorship in 1992, directing the bulk sample mining and supporting drill programs. In 2002 Fairfield merged with Almaden, which became the operator. All exploration programs from 1986 to 2005 were supervised by Wojtek Jakubowski, P.Geo. From 2006 to 2010 exploration was directed by Morgan Poloquin, P.Eng.

GMMC initiated exploration in June 2011 while completing a purchase agreement with Almaden. Diamond drilling plus pole-dipole IP and Titan 24 geophysical surveys were conducted in 2011 under the supervision of Peter Theirsch, P.Geo. From 2012 to 2015 diamond drilling plus pole-dipole IP geophysical was conducted under the supervision of Robert G. Wilson, P.Geo. Bulk sample open pit mining in 2012 and 2014 was supervised by Jim Balmer, P.Geo.

Section 6 has already detailed the exploration that occurred on the Property. It should be noted that historical soil sampling was conducted in stages within individual grids covering the various zones on the Property. As such, each grid had a separate baseline and grid numbering system. In the days before GPS, compassed grid lines tended to wander depending on the magnetic properties of the local bedrock and the skill set of those establishing the grid. In reviewing grid maps for the South Showing it was noted that actual grid stations for some lines were up to 30 m away from theoretical locations. The South Showing grid map is the only one constructed with sufficient control to allow identification of the discrepancies. All other grid maps were constructed as if actual grid stations were at the theoretical locations. With this limitation, best efforts were made in the early 2000s to reconcile the grid and sample locations to a Mine Grid system. The Mine Grid stations were subsequently mathematically transformed to UTM Grid coordinates. Hence soil sample stations possessing UTM coordinates are considered approximate, not absolute. Adding to the complication of interpreting soil sample results is the fact that this area was glaciated, with last apparent flows from NW to SE. The soils tended to smear in the direction of glacial movement and hence the soil anomalies are likewise smeared to the SE. While single veins would tend to have a hard up-ice soil anomaly boundary, the Property has multiple veins occurring within the smear zone, hence these boundaries are not as clear.

Despite the limitations noted, soil sampling was used successfully to target trench locations, which in turn allowed the discovery of gold-mineralized quartz veins in the trenched bedrock, several of which

were tested at depth by diamond drilling. Several gold-mineralized zones other than Siwash North have been discovered on the Property, including Bullion Creek North and South, Siwash East, Gold Creek East and West, Lake, End/Great Wall, Discover (North Showing), South Showing, Elusive Creek, and the Agur Option area. The previous NI 43-101 PEA report on the Elk Gold Project by Pooley et al. (2011) discusses these zones in detail.

9.2 Airborne Geophysical Survey

In June 2019 a helicopter geophysical survey was flown over the central portion of the Property for GMMC. The survey was flown to identify the magnetic and radiometric signature of each of the known gold-mineralized zones to assist in planning future exploration programs in areas of significant glacial drift cover. Precision GeoSurveys Inc. flew the survey, with subsequent interpretation by Campbell & Walker Geophysics Ltd. The following sections are drawn from C. Campbell's assessment report (Campbell, in preparation).

9.2.1 Acquisition and Processing

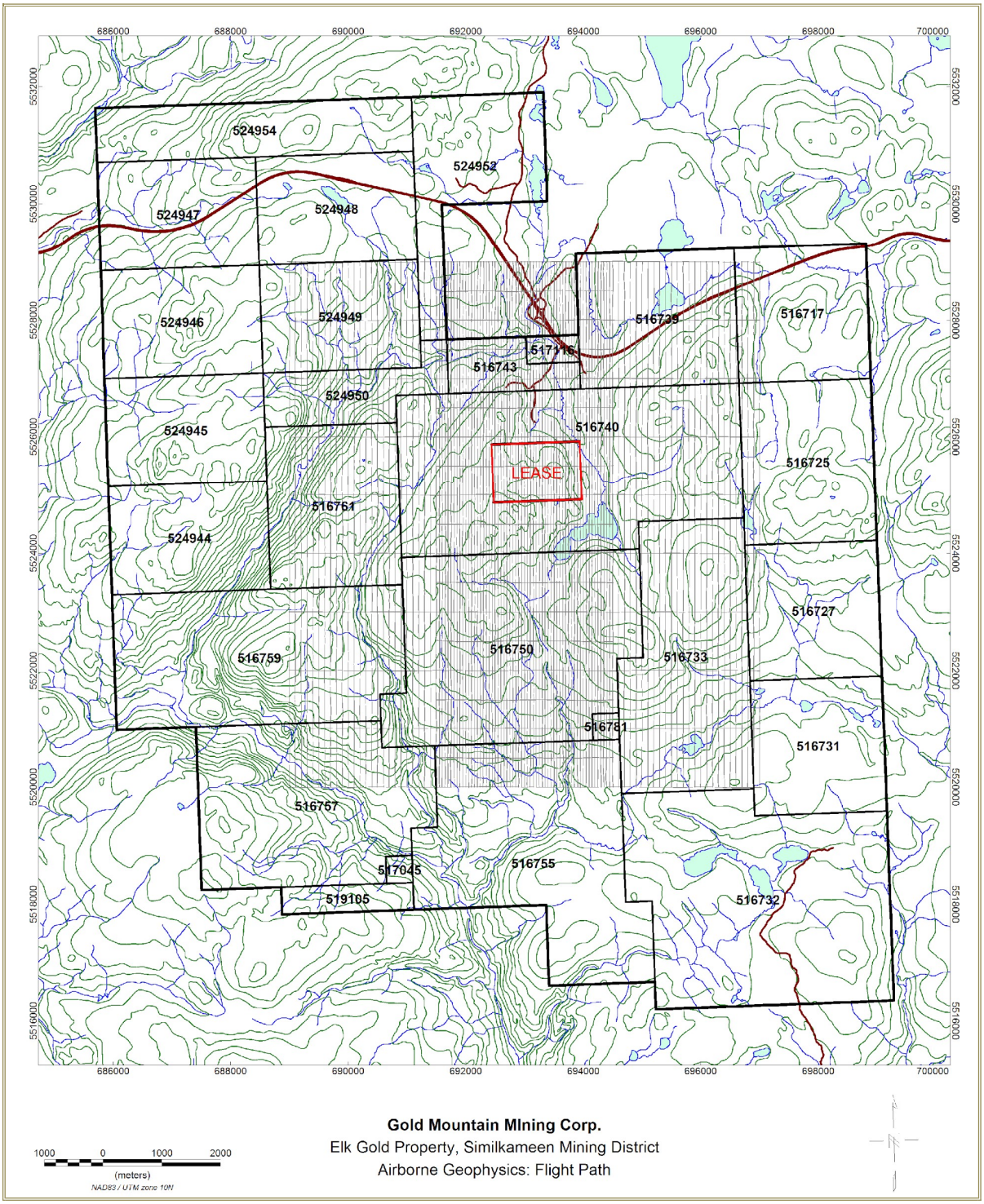
Precision GeoSurveys Inc. flew a high-resolution helicopter-borne magnetic and gamma-ray spectrometry survey for GMMC over a portion of the Property. The airborne geophysics was undertaken and completed in three days of flying in mid-June 2019. The survey coverage was 1,107.691 line km spread over 7,233.875 ha, and comprised north–south traverse lines spaced at 100 m, tied by east–west tie or control lines at 500 m spacings (Figure 9-1). The central portion of the survey block was "filled-in" with effective 50 m spaced traverse lines.

The base of operations for this airborne program was the municipal airport at Merritt, about 45 km northwest of the survey block's centre. Data was acquired using an Aerospatiale AS350-BA helicopter (registration C-GSVY) at a height of 40 m mean terrain clearance.

Projection and datum specifications for all maps:

Map projection.....NUTM10
Datum.....WGS84
Central meridian .....123° west
False Easting.....500,000 m
False Northing .....0 m
Scale Factor .....0.9996 m

The aeromagnetic data were subjected to standard heading, lag, and diurnal corrections; tie-line levelling and micro-levelling were then applied, followed finally by an International Geomagnetic Reference Field (IGRF) correction to derive a residual magnetic intensity before the reduction-to-pole transfer. A calculated vertical and horizontal gradient based on the RTP magnetics have additionally been generated.



Source: Precision GeoSurveys, June 2019.

Figure 9-1: Airborne Geophysics Flight Path with Active Claims

Radiometric data was similarly subjected to industry-standard and accepted corrections; these included calculation of effective height, lag correction, aircraft and cosmic background corrections, radon background correction, stripping ratios, attenuation corrections, and, finally, conversion to apparent radioelement concentrations.

9.2.2 Data Presentation and Imaging

The digital data were gridded after standard processing and corrections. Representative images are listed below and shown in the following pages:

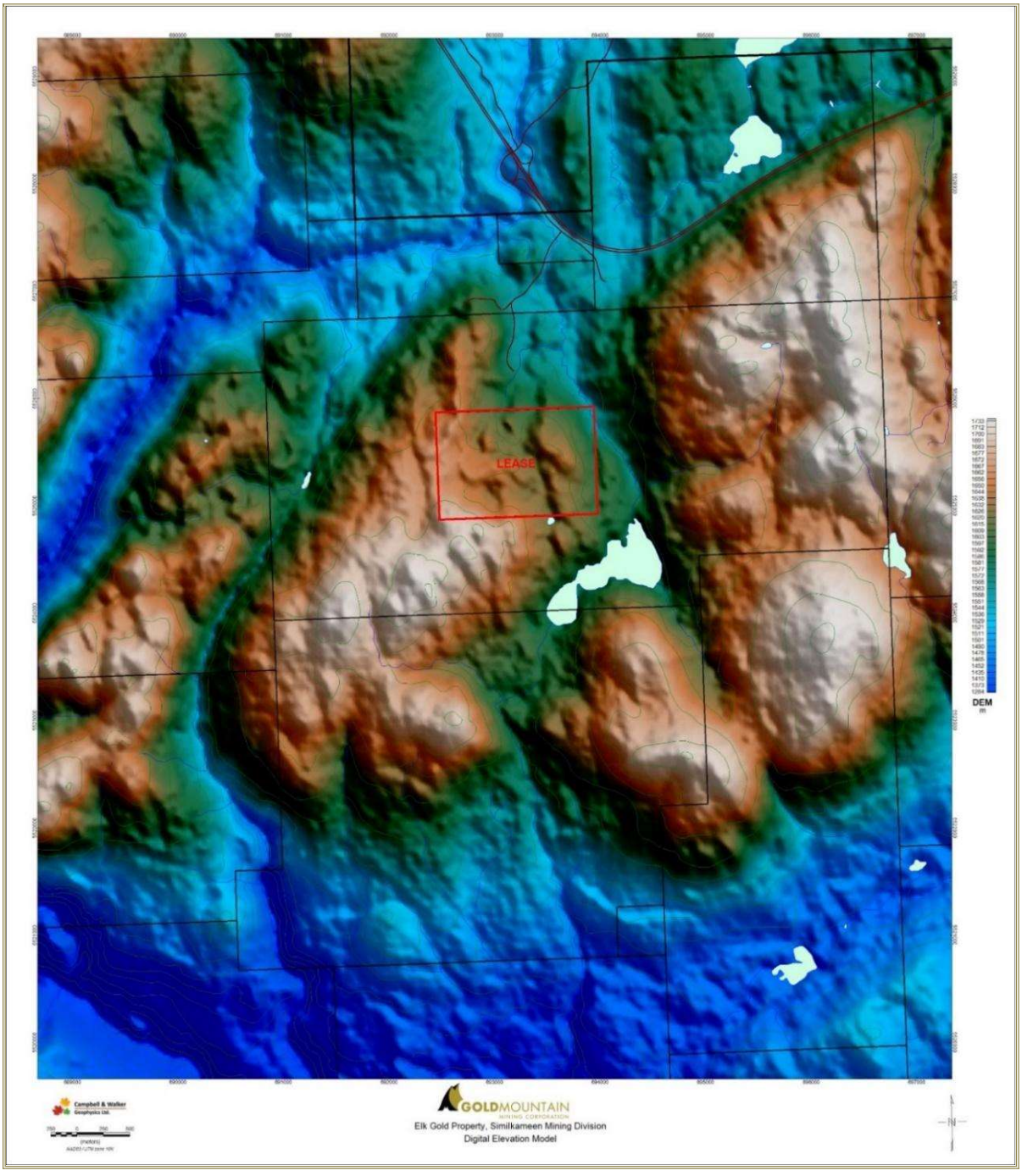
- Digital terrain model (DTM) (Figure 9-2)
- Residual magnetic intensity (RMI)—removal of IGRF from TMI (Figure 9-3)
- Total counts—exposure rate (TCexp) (Figure 9-4)
- Potassium—equivalent concentration (% K) (Figure 9-5)
- Thorium—equivalent concentration (eTh) (Figure 9-6)
- Uranium—equivalent concentration (eU) (Figure 9-7)
- Ternary map (TM) (Figure 9-8).

Note well: the RMI grid, and subsequently all further transform and derivative magnetic grids discussed in this interpretation report are based on the Intrepid Geophysics' INTREPID software variable density gridding algorithm. The purpose of this method is to estimate various wavelength contents optimally, and to combine them into one, final grid that has the best assemblage of observed wavelengths from the datasets provided. This approach was used to best deal with the two different traverse line spacings.

Additional interpreted gridded data are presented in Section 9.3, including:

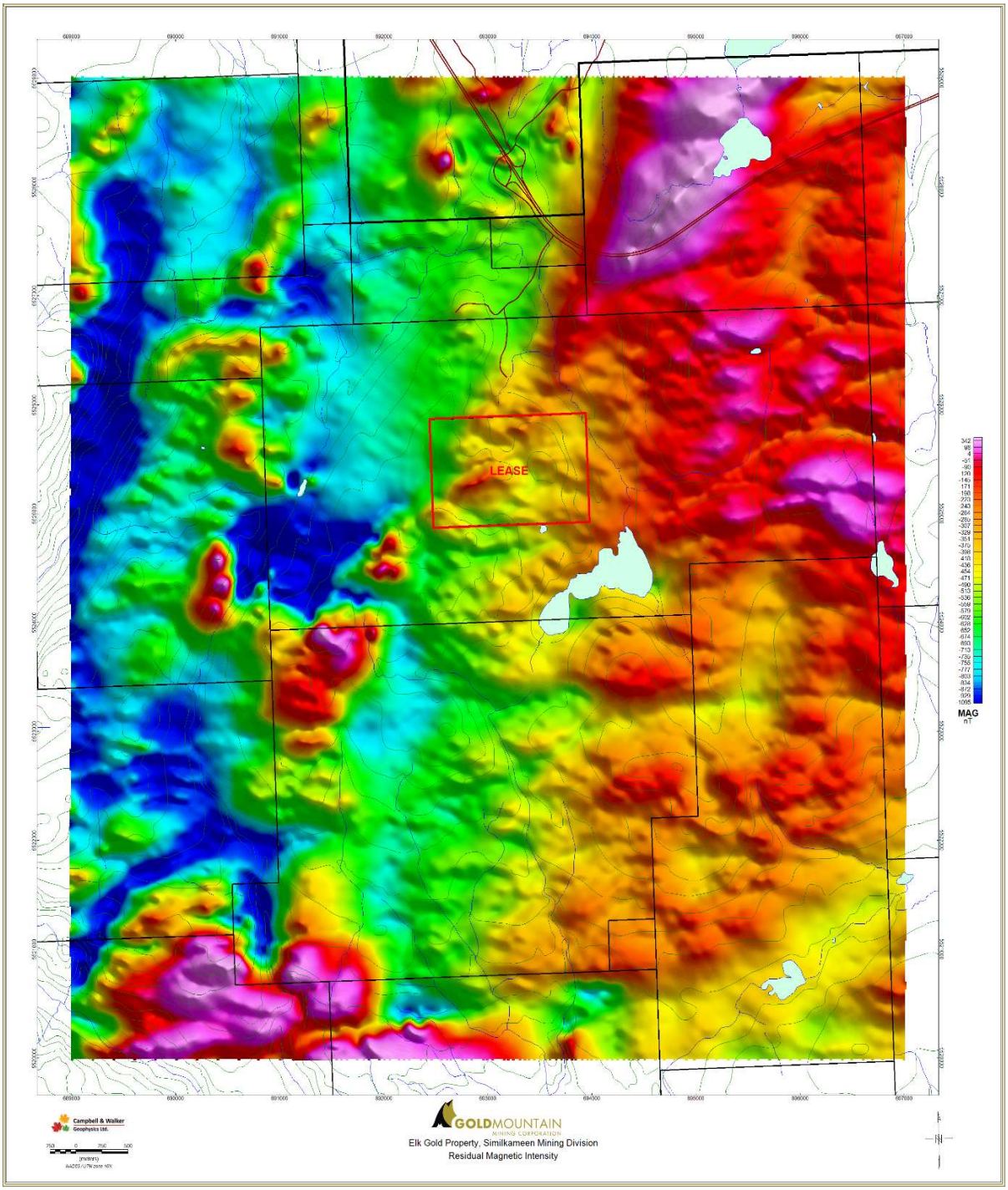
- Total magnetic intensity (TMI) (Figure 9-9)
- Reduced to magnetic pole (RTP)—reduced to magnetic pole of RMI (Figure 9-10)
- Calculated horizontal gradient (CHG)—total horizontal gradient of RMI (Figure 9-11)
- Tilt derivative (TD)—arctan of the first vertical derivative/total horizontal derivative (Figure 9-12)
- Tilt derivative—positive phase only (Figure 9-13)
- Analytic signal (total gradient) (Figure 9-14)
- Edge zone filter (Edge Group) (Figure 9-15)
- ZS area filter (Block Group) (Figure 9-16)
- Multi-scale edge detection 'worms' overlain on RTP Magnetic Intensity (Figure 9-17)
- CET grid analysis: contact occurrence density with 1 & 2 standard deviation (SD) above means (Figure 9-18)

- CET grid analysis: orientation Entropy with 1 & 2 SD above mean (Figure 9-19)
- Low Th/K areas of interest (Figure 9-20).



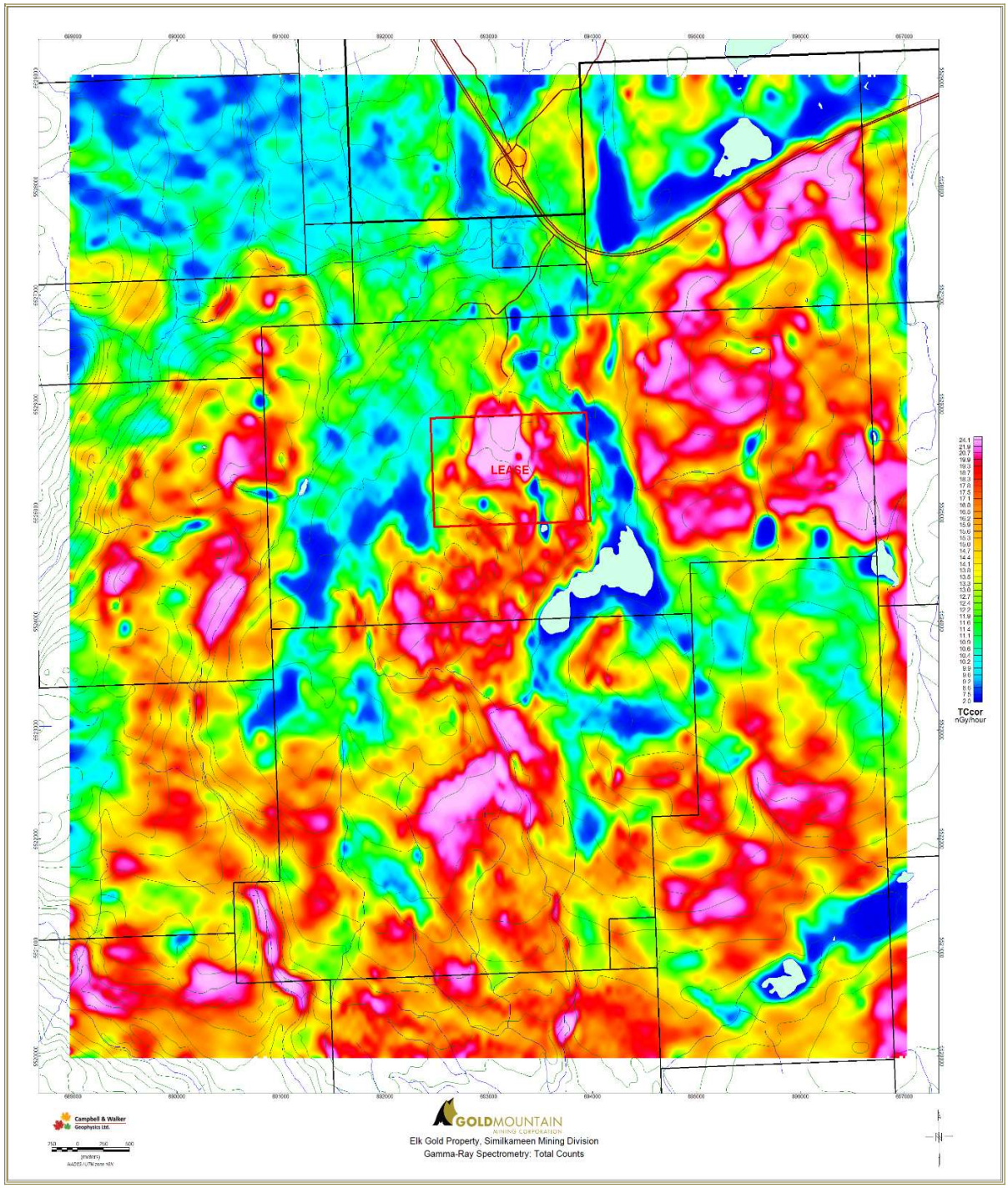
Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-2: Digital Terrain Model



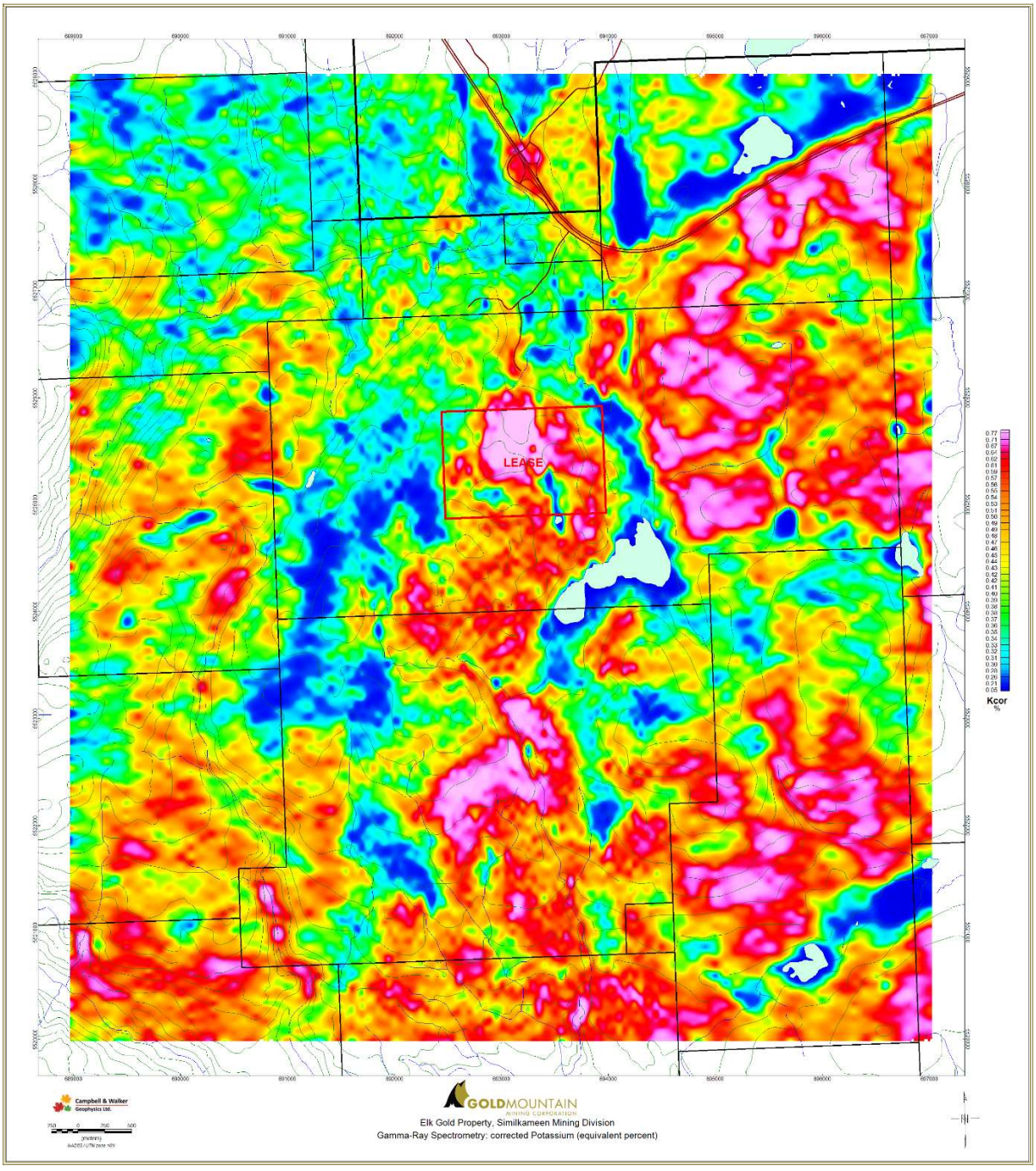
Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-3: Residual Magnetic Intensity (IGRF Removed from the Measured Total Magnetic Intensity)



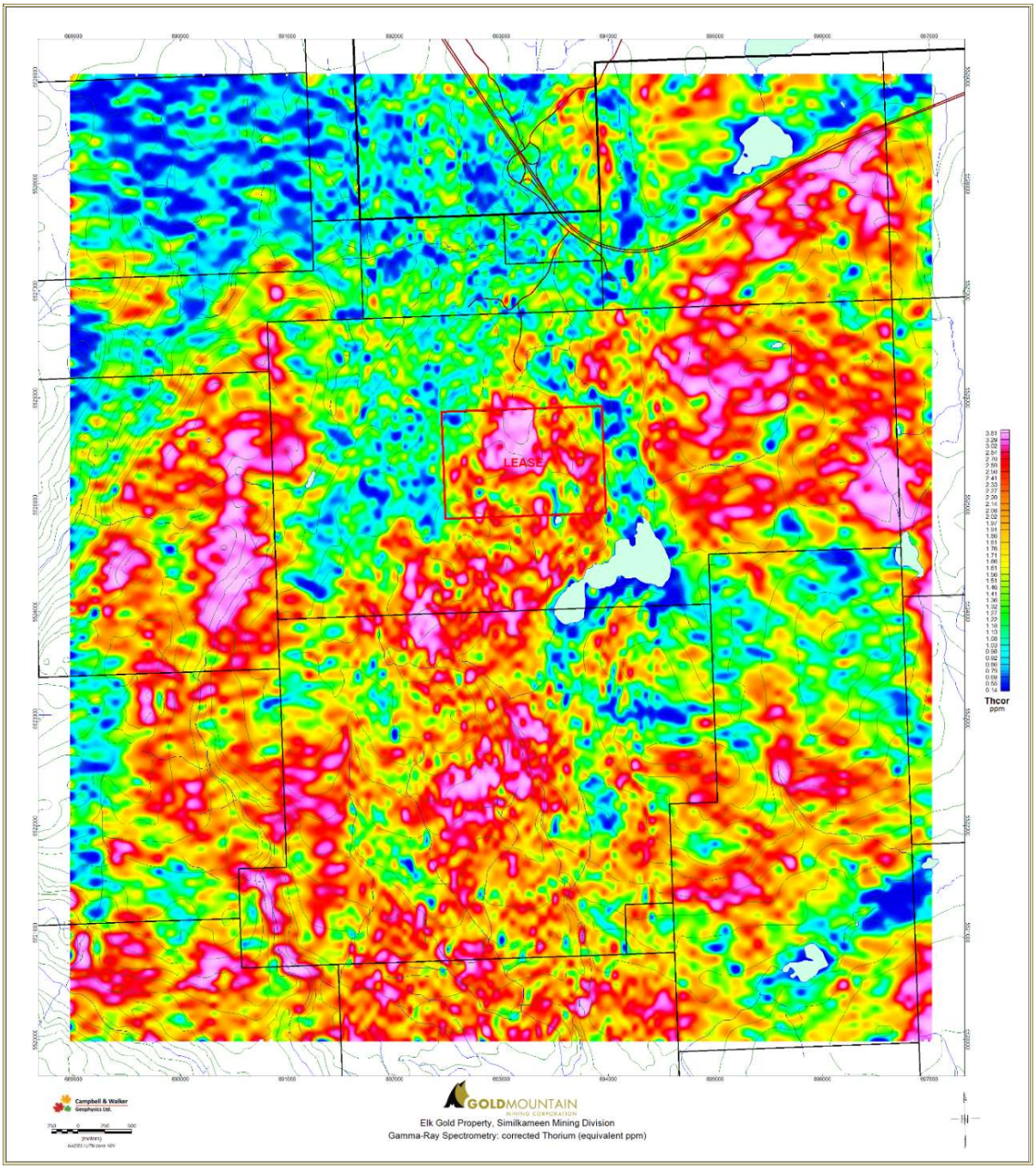
Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-4: Total Counts (Natural Exposure Rate)



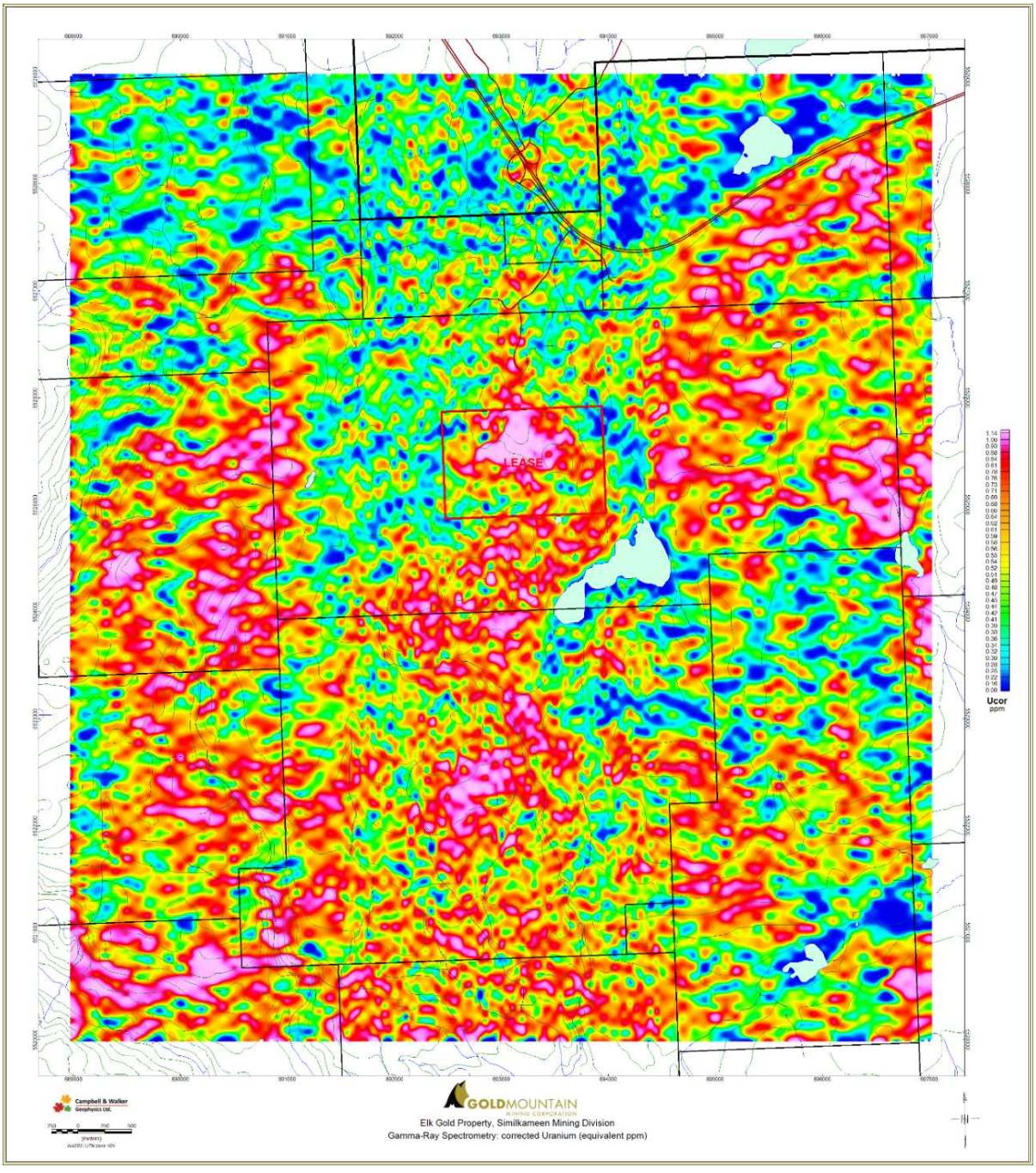
Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-5: Potassium (Equivalent Concentration)



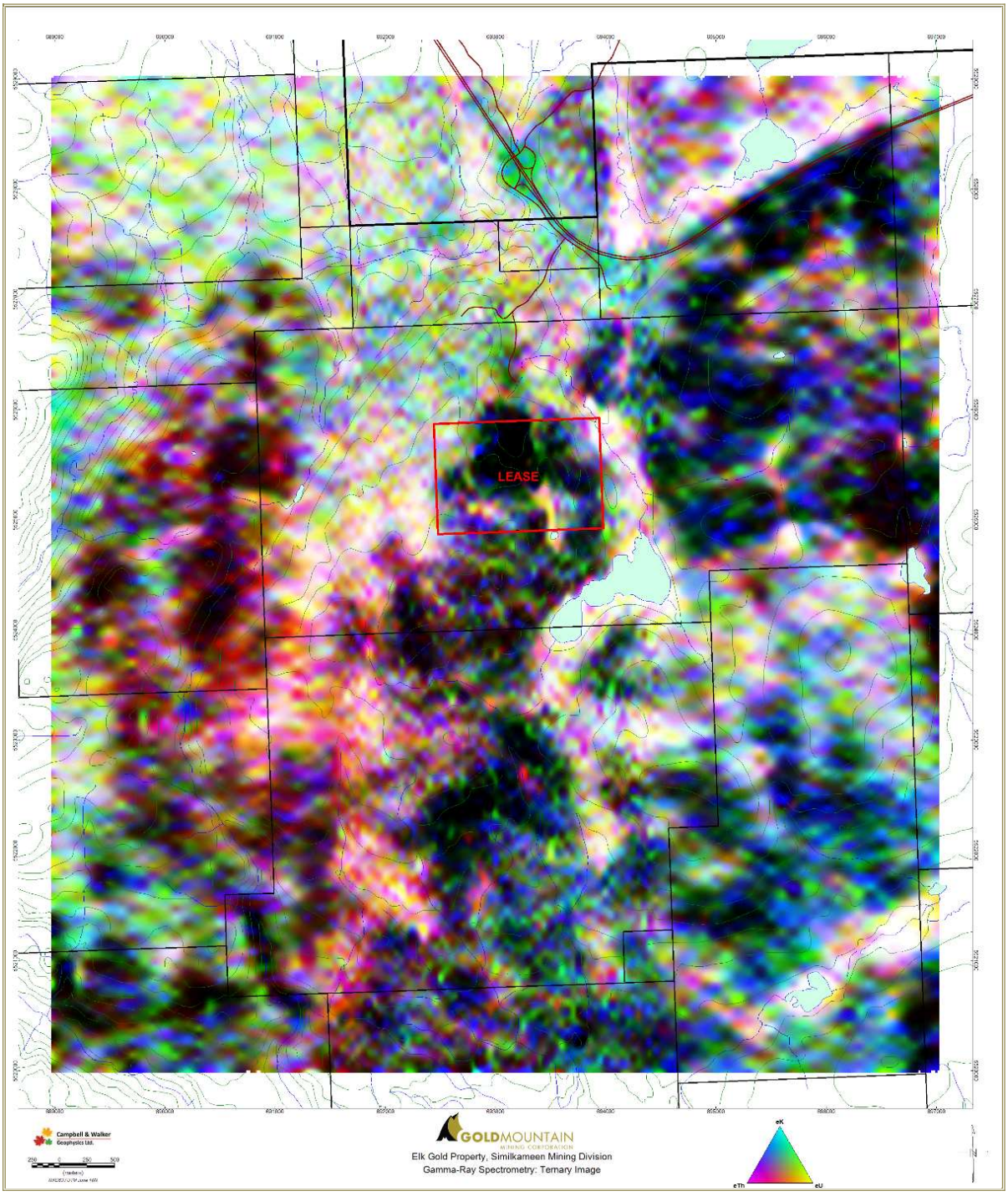
Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-6: Thorium (Equivalent Concentration)



Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-7: Uranium (Equivalent Concentration)



Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-8: Ternary CMY Image (Kpercent_eTh_eU)

9.2.3 Interpretation

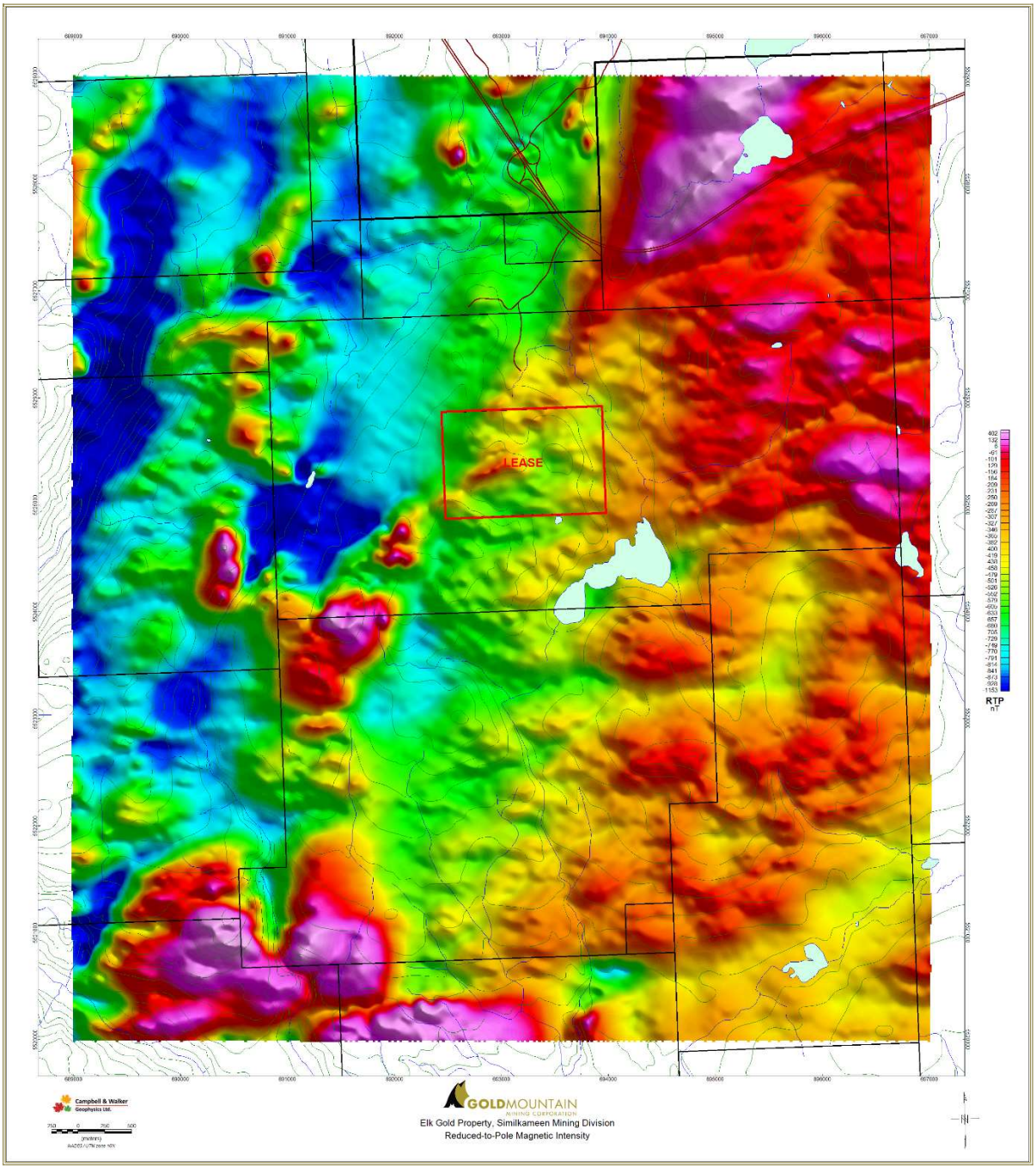
The aeromagnetic grids and images were examined especially with a view to delineating and extracting indications of possible structural fabric, on the premise that structural analysis plays a key role in resource exploration. Almost all styles of mineral deposits and hydrocarbon accumulations are controlled either directly or indirectly by some form of structural focus.

The reduced-to-pole (RTP) grid (Figure 9-9) often, if not normally, comprises the basis for these analyses. RTP is a fundamental process that, in most situations, yields imagery representing the geometry of magnetic rock units better than TMI data. Its physical basis is sourced in potential field theory (Baranov & Naudy, 1957), and where it can be successfully applied numerically it is considered a "must" for the interpreter; in essence, the dipolar nature of magnetic anomalies is removed, and peak RTP magnetic values relate more closely to the centre of magnetic rock bodies; asymmetries in the RTP imagery more closely reflect true dips and plunges (Isles & Rankin, 2013).

The RTP process mathematically moves the measurement location to the (commonly north) magnetic pole; i.e., the transform shifts anomalies to be directly over their source, facilitating an easier interpretation. The Earth's field here has mean declination = 16.1° and inclination = ~71.2°.

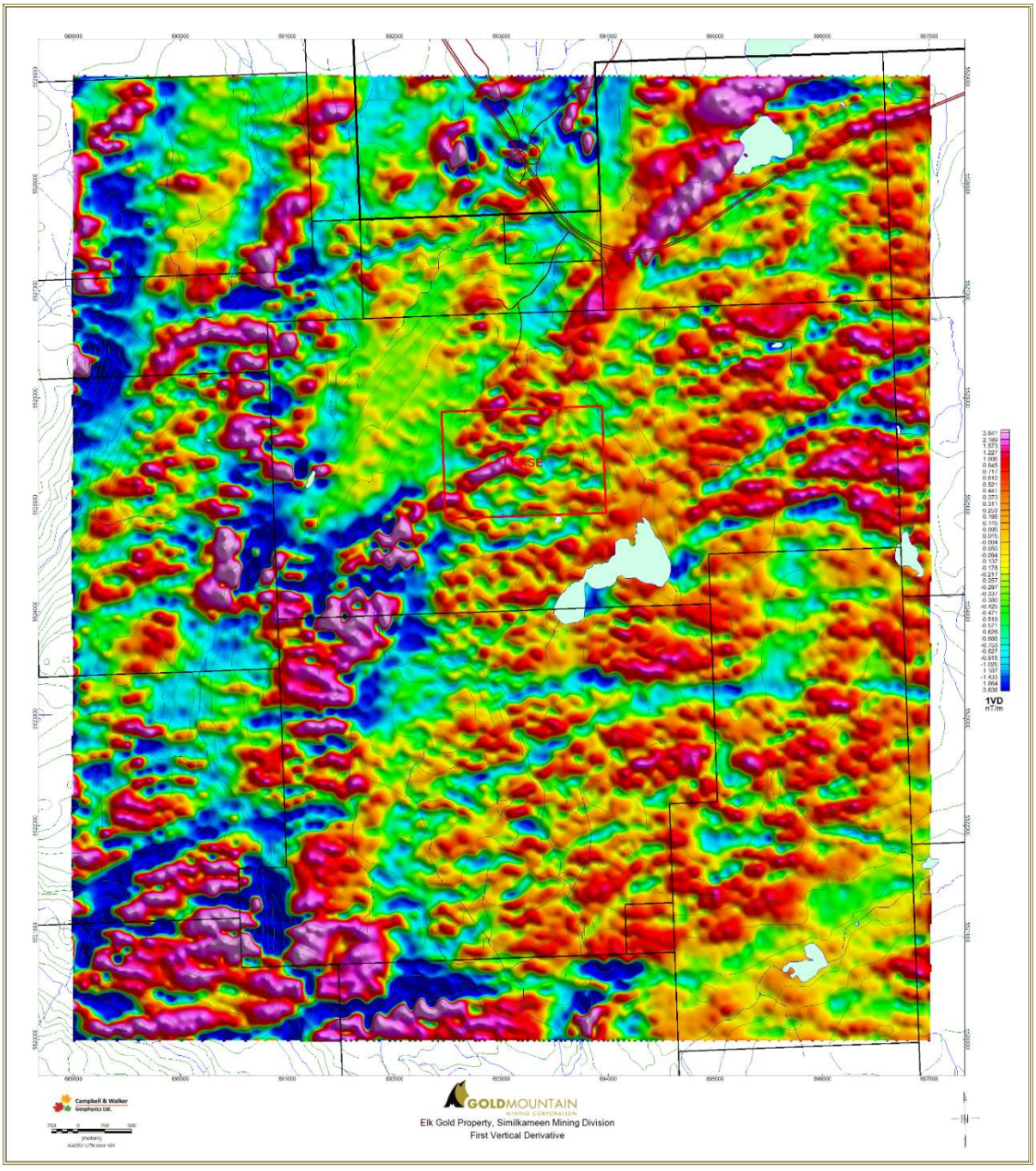
The goal of RTP is to simplify magnetic maps—essentially making them appear to be more like gravity maps. In practice, RTP is complicated because anomalies may be rotated by tectonism—rocks of different ages may be reverse or normally polarized—and related factors. A further complication arises as the essentially two-dimensional (2-D) fast Fourier transform (FFT) is being applied here in an area of significant topographic relief (i.e., about 500 m), inherently non-2-D, and thus non-conforming to the underlying assumptions of a 2-D Fourier operator. Although the derivatives in this study are all based on the RTP magnetic intensity, some caution as to precise locations of specific features should be exercised. However, as the mean inclination of the Earth's field on the Elk Gold Project is ~71.2° (considered to be a 'moderately high' inclination), large displacements of anomalies are not anticipated.

The RTP magnetic intensity was passed through several derivatives and filters; selected images are shown in the figures below. The first vertical derivative (1VD) (Figure 9-10) is a common high-pass filter; both north-northwest and northeast linear features are accordingly mapped. The total horizontal derivative (THD) (Figure 9-11) yields further information, with a maximum on both flanks of a feature as opposed to the 1VD single maxima.



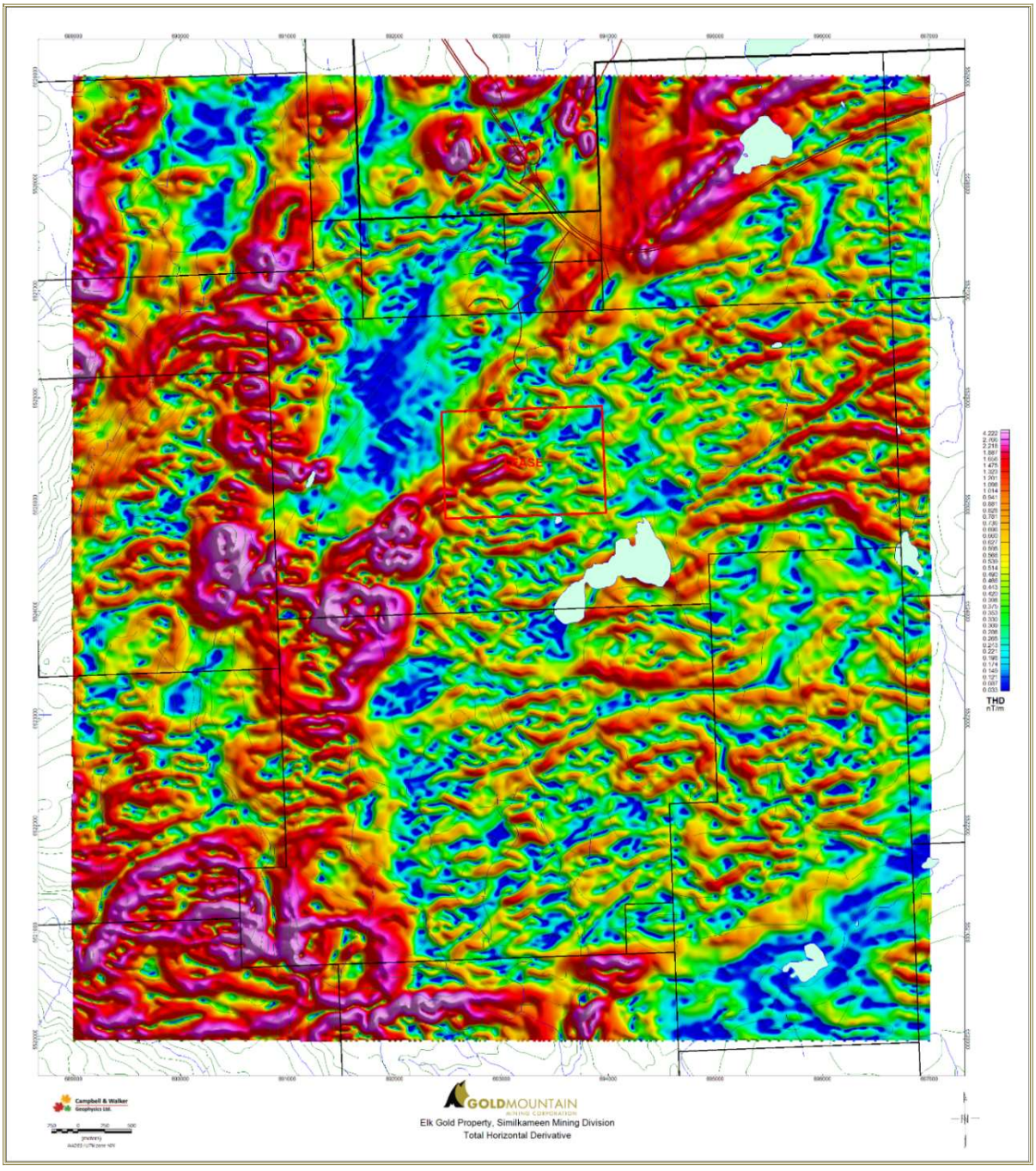
Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-9: Reduced-to-Pole Magnetic Intensity



Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-10: First Vertical Derivative

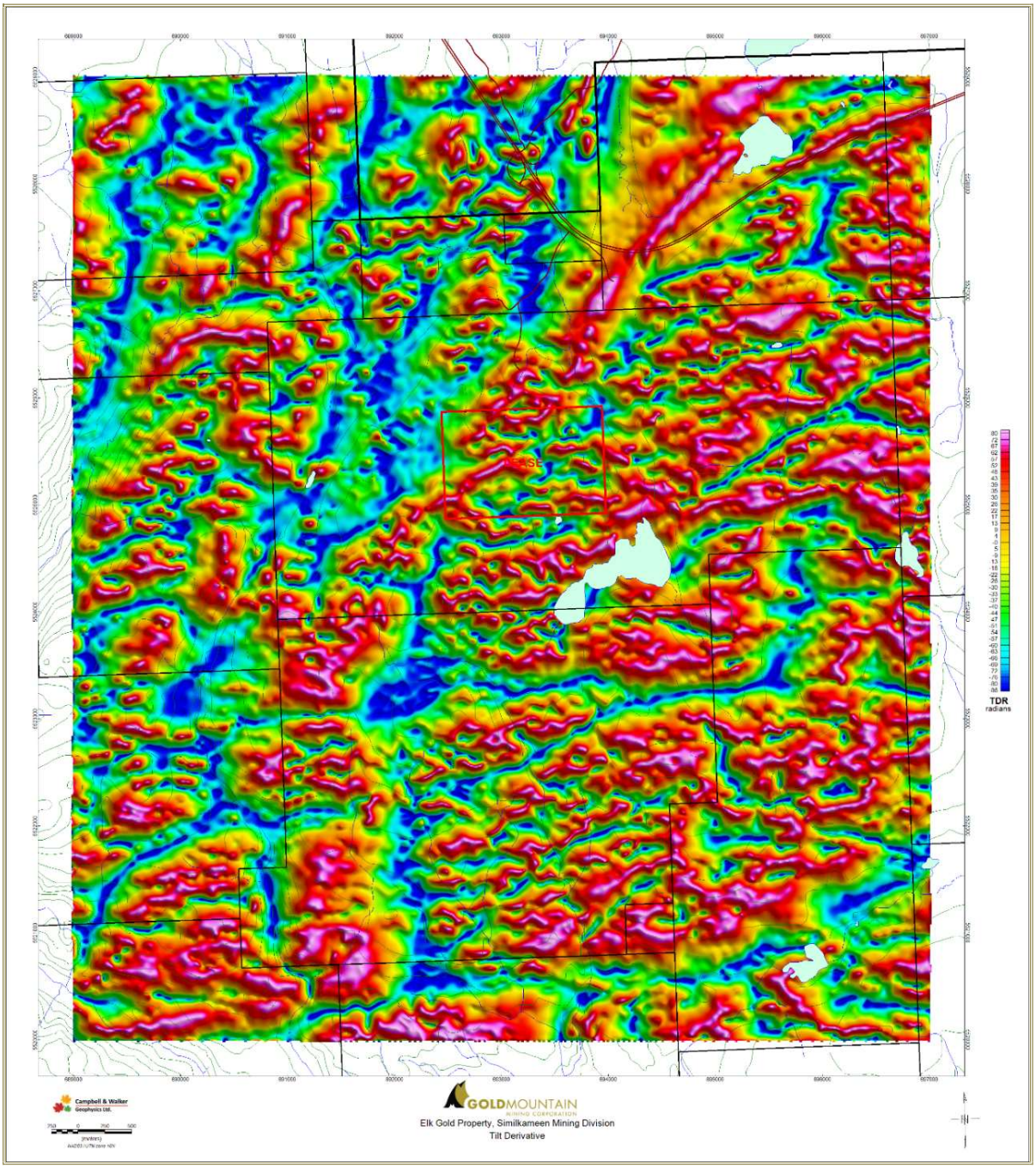


Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-11: Total Horizontal Derivative

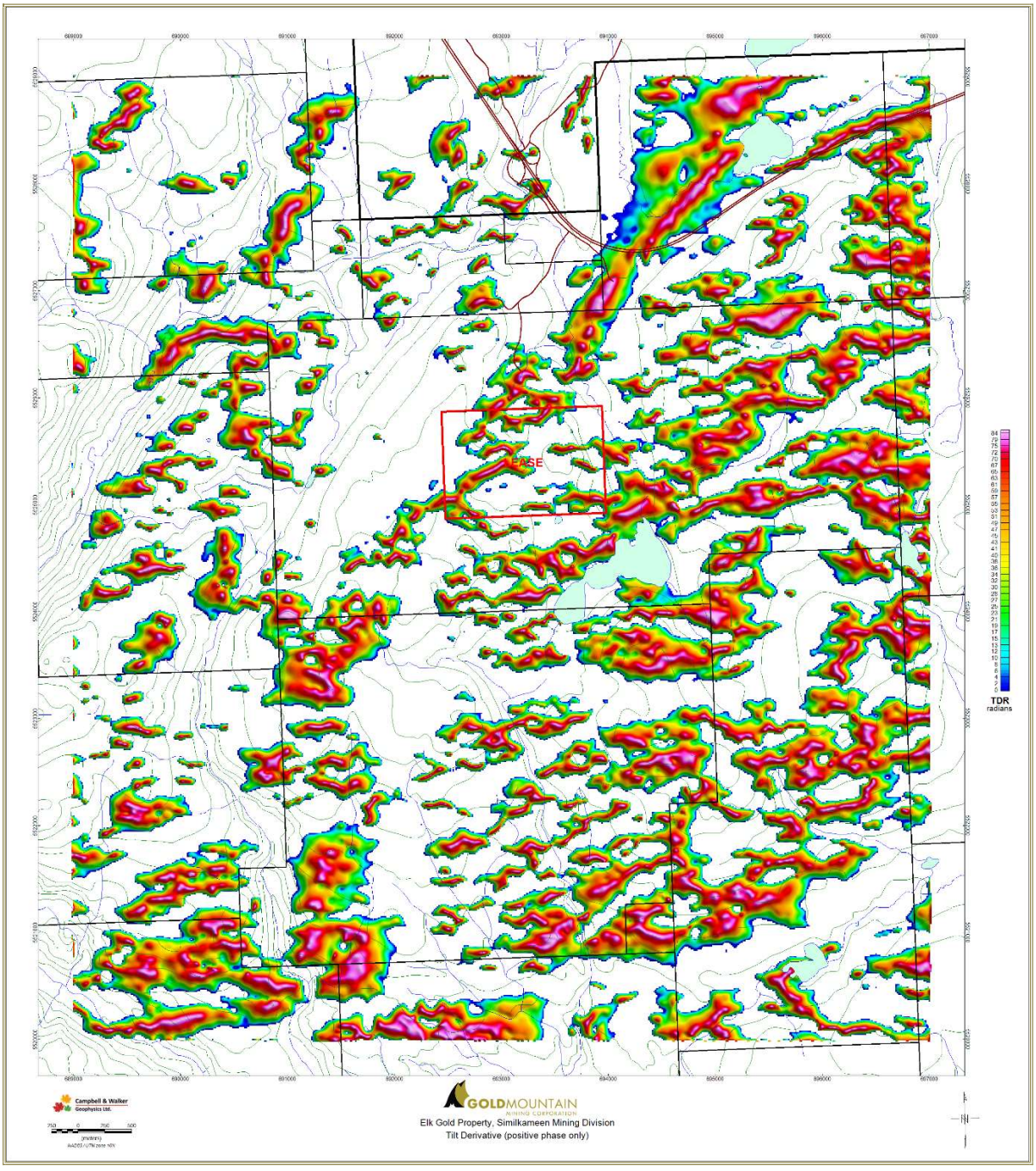
Another common derivative, or filter, is the tilt derivative residual (TDR) (Miller and Singh, 1994); this is the arctan of the 1VD/THD (Figure 9-12). Because it is derived from a ratio, it has a normalized data intensity range. The TDR produces patterns similar to the 1VD and THD grid images, but portrays

responses from deeper and shallower sources with similar amplitudes, and may therefore be confusing. The positive phase only of the TDR is also shown (Figure 9-13), helping to emphasize positive magnetization.



Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-12: Tilt Derivative

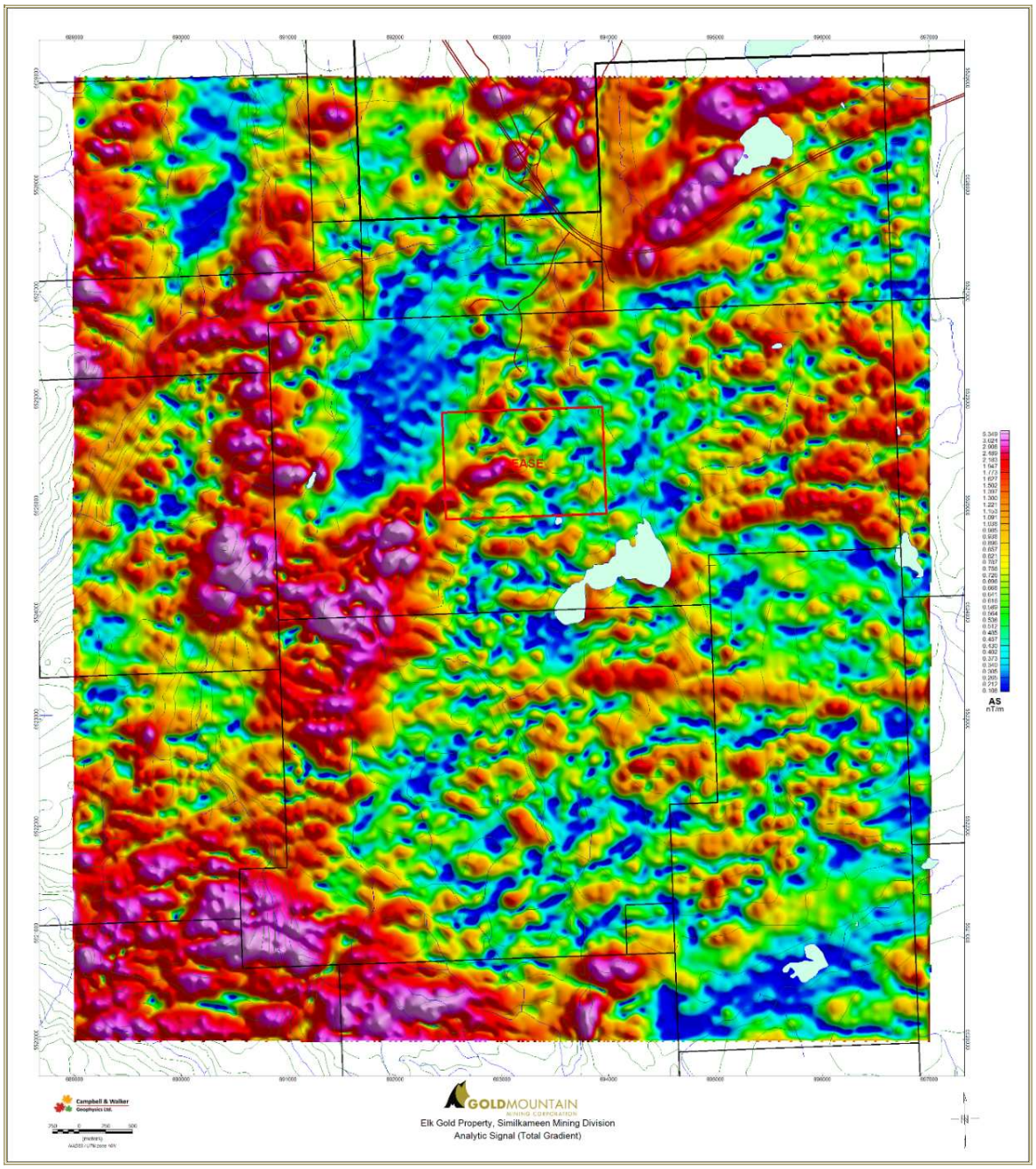


Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-13: Tilt Derivative (Positive Phase Only)

The analytic signal (AS), or total gradient, is run on the RMI, as the process is relatively immune to the direction of Earth's magnetic field, and thereby serves as a check on the RTP for remanent magnetism (Figure 9-14). The AS produces highs over magnetic sources. A comparison of the AS and RTP suggests

that the bulk of magnetization observed in this study is induced; however, an exception to this is found about 2.5 km southeast of the mining lease, where a magnetic anomaly under the small lake (Galena Lake) exhibits remanence as does the east–west trending dyke immediately south of that same lake.

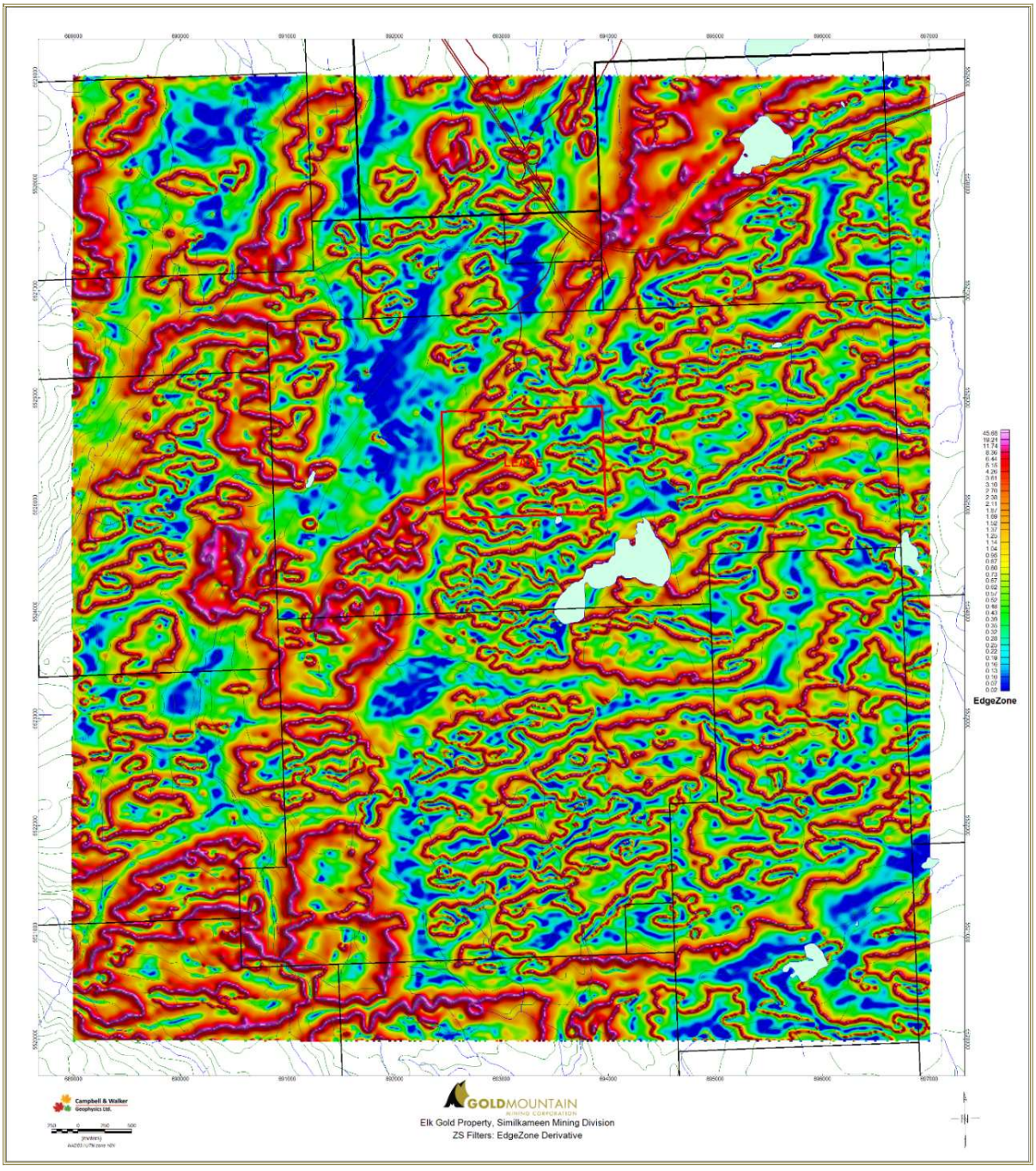


Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-14: Analytic Signal (Total Gradient)

9.3 New Enhancement Filters (Shi & Butt, 2004)

Besides the standard filters such as 1VD, THD, TDR, and AS, new enhancement filters for geological mapping have been developed for the purpose of enhancing weak magnetic anomalies from near-surface sources while simultaneously enhancing low-amplitude, long-wavelength magnetic anomalies from deep-seated or regional sources. That suite of filters known as 'ZS' filters' (after Zhiqun Shi, the primary author of this development). The Edge filter group (Figure 9-15) highlights edges surrounding both shallow and deeper magnetic sources. The results are used to infer the location of the boundaries of magnetized lithologies.

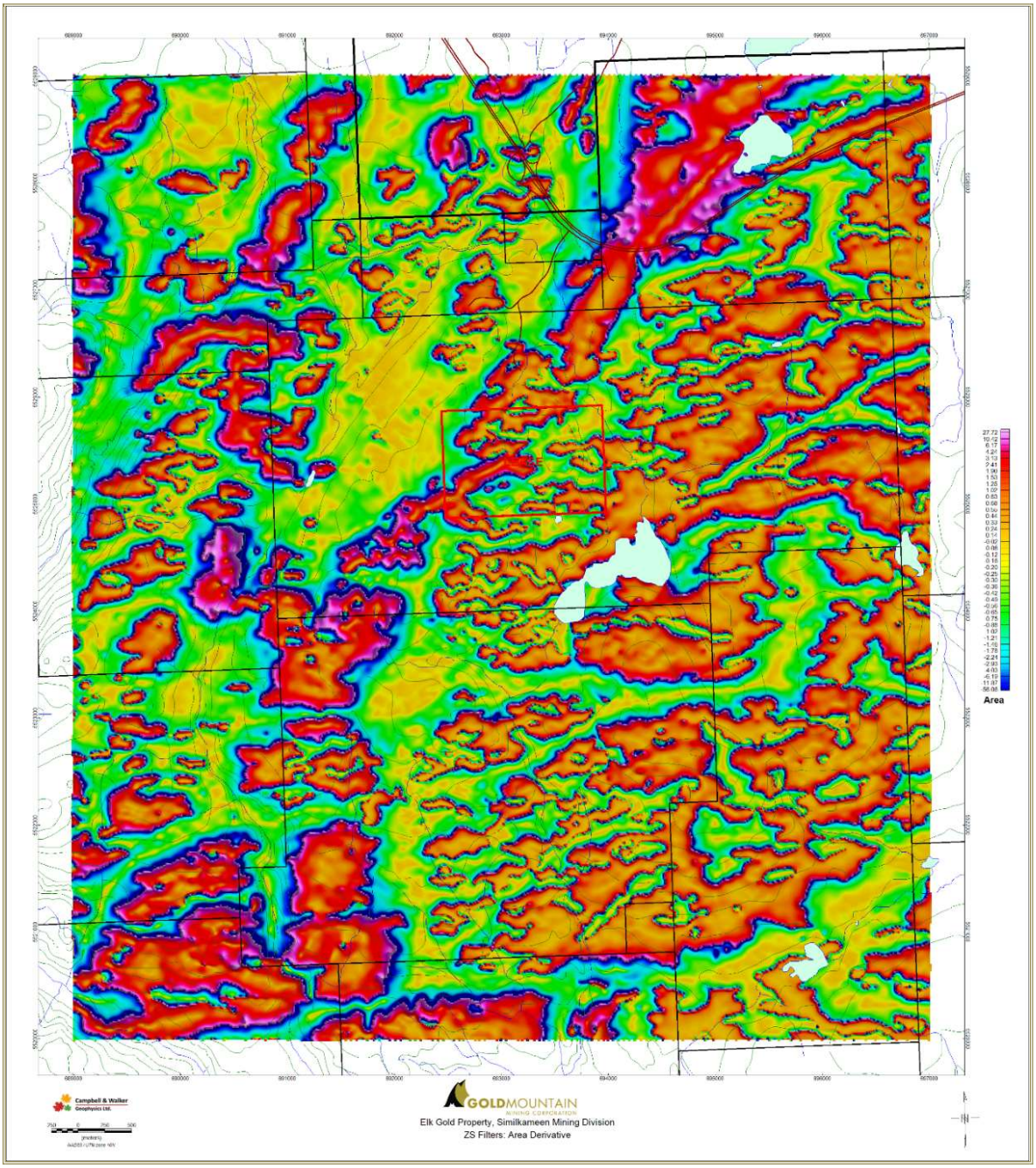


Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-15: ZS Edge Zone Filter (Edge Group)

The Block filter group (Figure 9-16) has the effect of transforming the data into "zones" that, similar to image classification systems, segregate anomalous zones into apparent lithological categories.

Both filter groups change the textural character of a dataset and thereby facilitate interpretation of geological structures.

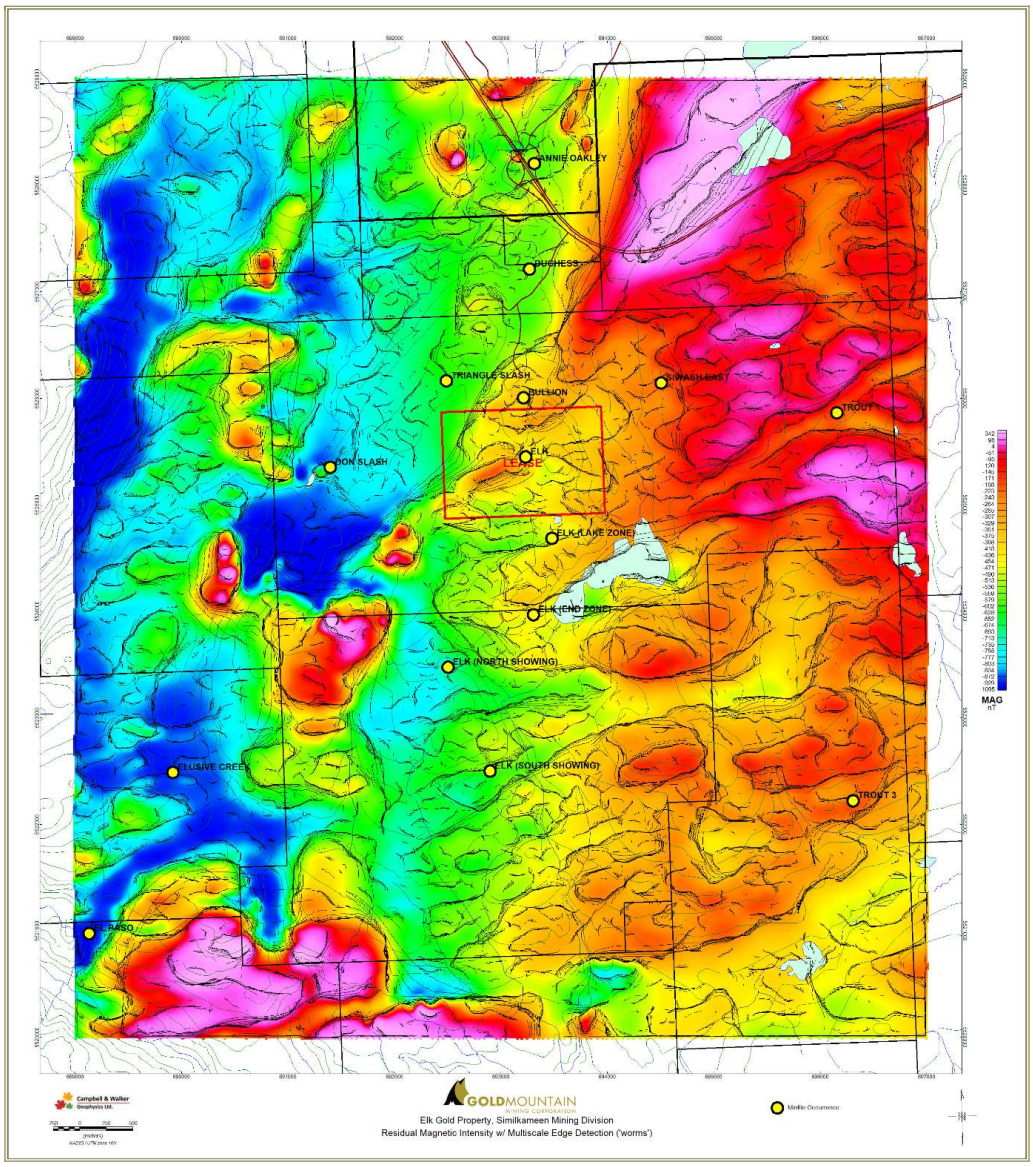


Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-16: ZS Area Filter (Block group)

9.4 Structural Analysis—Multiscale Edge Detection (Worms)

A two-pronged approach was taken to identify and map magnetic "fabric"—multi-scale edge detection (also known as "worms") as described by Milligan et al., (2004) and a fractal analysis as described by Holden et al. (2008). Current implementation (INTREPID software) of multi-scale edge analysis includes the use of Eüler worms, which provide a view of structural geology obtained directly from potential field geophysical data. The method is based on Fourier techniques for continuation, RTP, and THD coupled with automatic edge detection (Figure 9-17).



Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-17: Multi-scale Edge Detection 'Worms' Overlain on RTP Magnetic Intensity

The analysis of lineaments is of fundamental importance to understanding geological structures and the stress regimes in which they are produced. Automatic analysis of lineaments has previously been done with information mapped from remotely sensed data, using either satellite-based imagery or aerial photographs. Potential field data may also be analyzed in terms of their lineament content. Edge detection and automatic trend analysis using gradients in such data are methods for producing unbiased estimates of sharp lateral changes in physical properties of rocks. The assumption is made that the position of the maxima in the horizontal gradient of gravity or magnetic data represents the edges of the source bodies, although this should be used with caution. Such maxima can be detected and mapped as points, providing the interpreter with an unbiased estimate of their positions. The process of mapping maxima as points can be extended to many different levels of upward continuation, thus providing sets of points that can be displayed in three dimensions, using the height of upward continuation as the z-dimension.

CSIRO in Perth substantially developed multi-scale edge detection in the late 1990s (Archibald et al., 1999). It combines wavelet analysis, potential field edge-detection and upward continuation to attempt to recover the horizontal location of property contrast interfaces, and to characterize their extension in depth. The INTREPID software implementation of this technique is shown below on the interpretation maps, Figure 9-18 and Figure 9-19. The combination of worms and pseudo-colour magnetic intensity) and derived derivatives of the magnetic intensity highlight structural fabric of the Property.

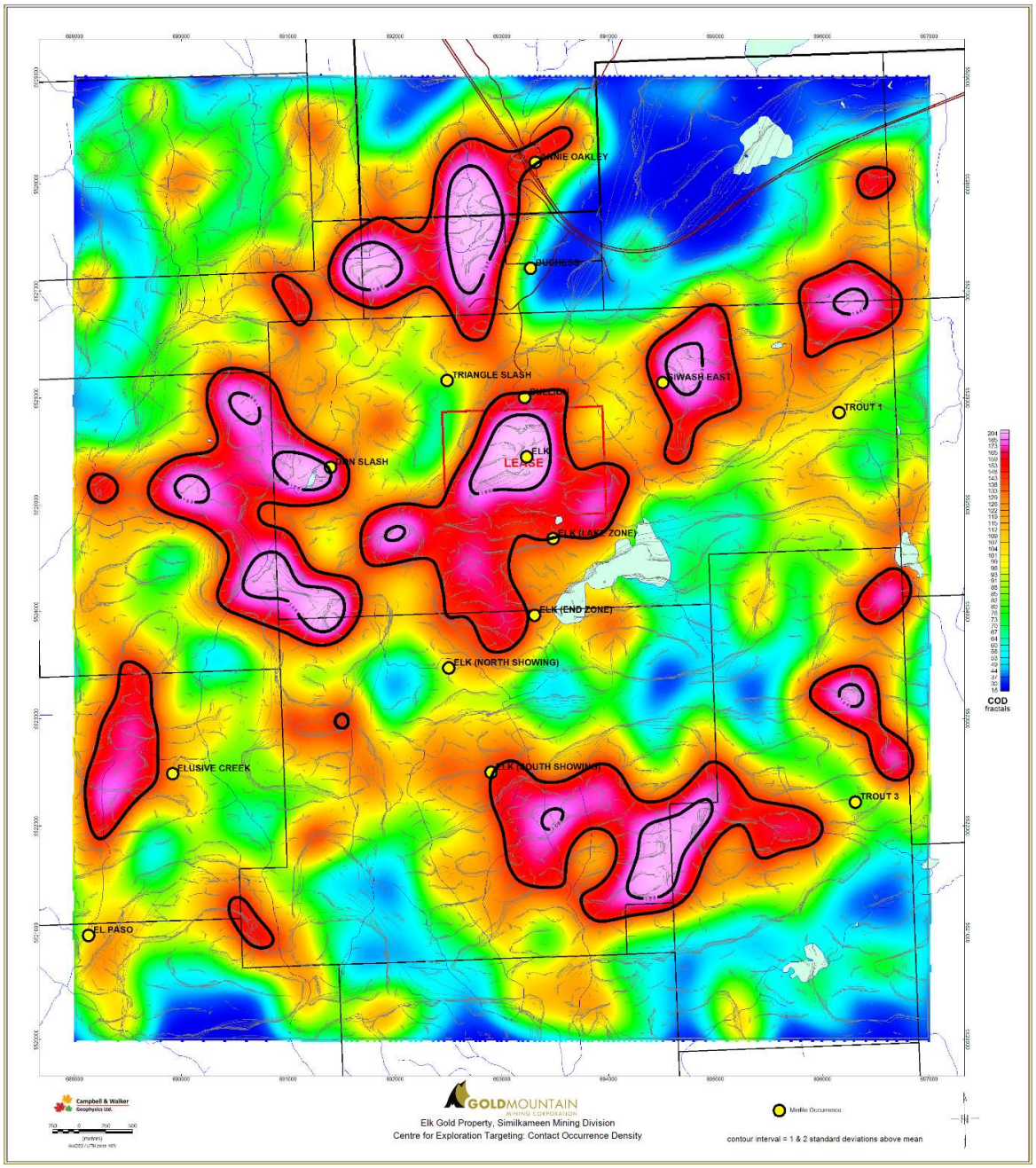
9.5 Structural Analysis—CET Grid Analysis

The Centre for Exploration Targeting (CET) grid analysis (via a Geosoft plugin) provides tools for texture analysis, lineation detection, and vectorization, leading to both structural complexity and orientation entropy “heat maps” shown on following images. Gold mineralization is known to occur near major crustal breaks manifesting as large-scale shear zones, which act as conduits for mineralizing fluids; mineralization occurs in regions of structural complexity adjacent to the shear zones. Progressing toward the automatic detection of such regions, the CET grid analysis system:

- Delineates magnetic discontinuity that corresponds to both lithological boundaries and shear zones using a combination of texture analysis and symmetry feature detection techniques
- Examines the data using fractal analysis to find areas nearby with a complex magnetic expression (zones of structural complexity)
- The most prospective areas are those where Inferred structural complexity occurs adjacent to the regions of magnetic discontinuity
- This approach may have merit in this instance, with the applicability on the Elk Gold Project in specific relation to the contact occurrence density and orientation entropy “heat maps.”

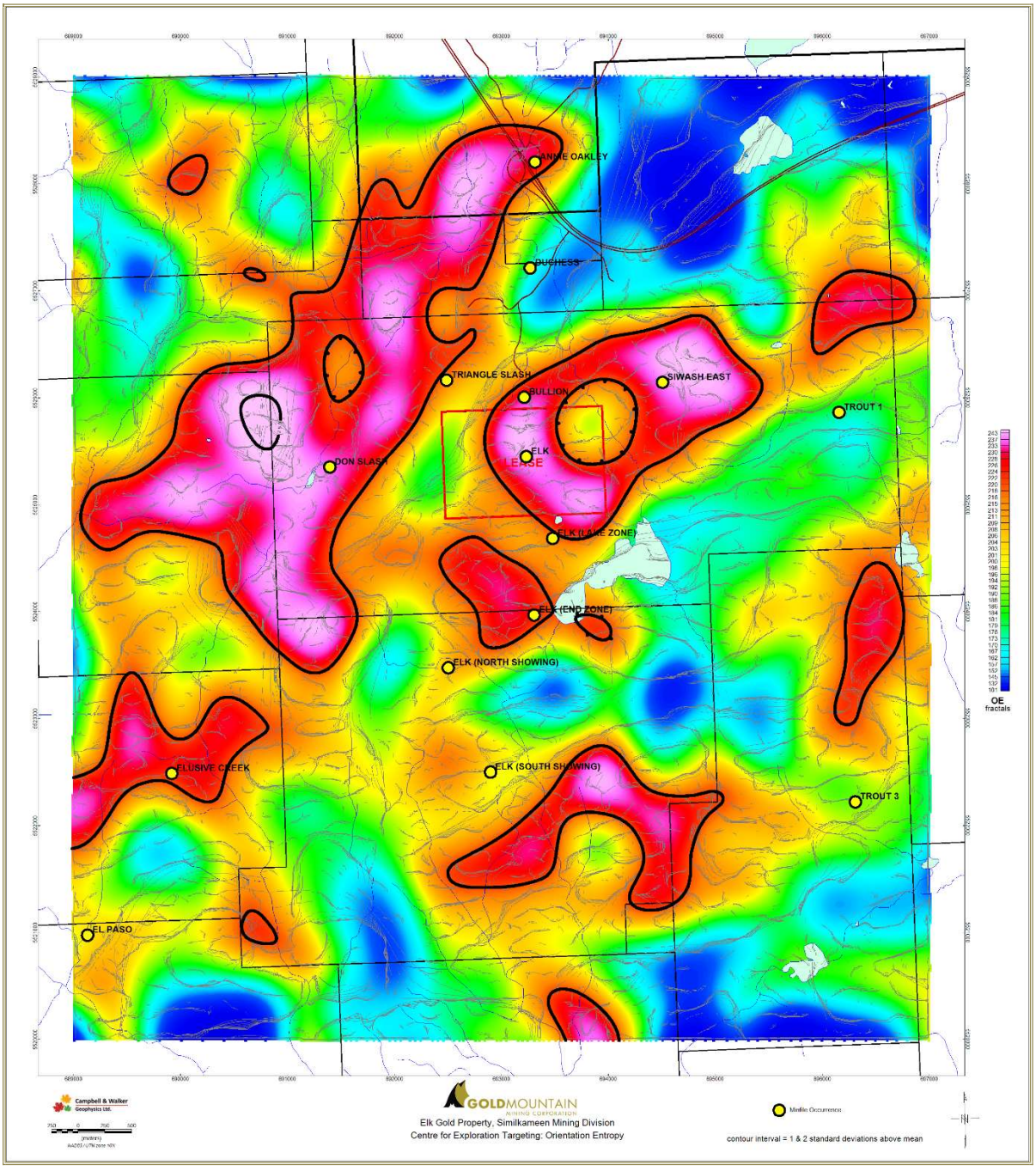
The contact occurrence density (COD) process generates a heat map that highlights high-density structural contacts, which include junctions and intersections of different structures. In contrast, the orientation entropy identifies those locations where structures have significant orientation changes.

Given an input of vectorized structures/trends (in this case, the multi-scale edge detection worms), pairs of line segments are analyzed to determine if they meet or intersect at an angle that is greater than a specified threshold. Each of these contact locations are then marked and used to cast a 2-D Gaussian vote towards the overall COD in the resulting heatmap.



Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-18: CET Grid Analysis: COD with 1 and 2 SD Above Means



Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-19: CET Grid Analysis: Orientation Entropy with 1 and 2 SD Above Mean

Particular zones of interest are delineated (somewhat subjectively, admittedly) by the 1 and 2 SD contours. However, the current resource (mining lease) is well-identified by this method, and similar zones may be observed north, west, and south. All are felt to be worthy of ground follow-up and

further investigation. These suggest complex zones having notable fractal intensity and, in turn, significant structural breakage.

The orientation entropy (OE) highlights regions where structures occur over a wide range of orientations, thus indicating areas of potential structural complexity. OE is based on the same fractal analysis/phase symmetry as the COD. The output grid comprises real values indicating the entropy of nearby structure orientations, centred about each cell. Regions of high entropy are considered to exhibit high statistical randomness, and thus areas of potential structural complexity.

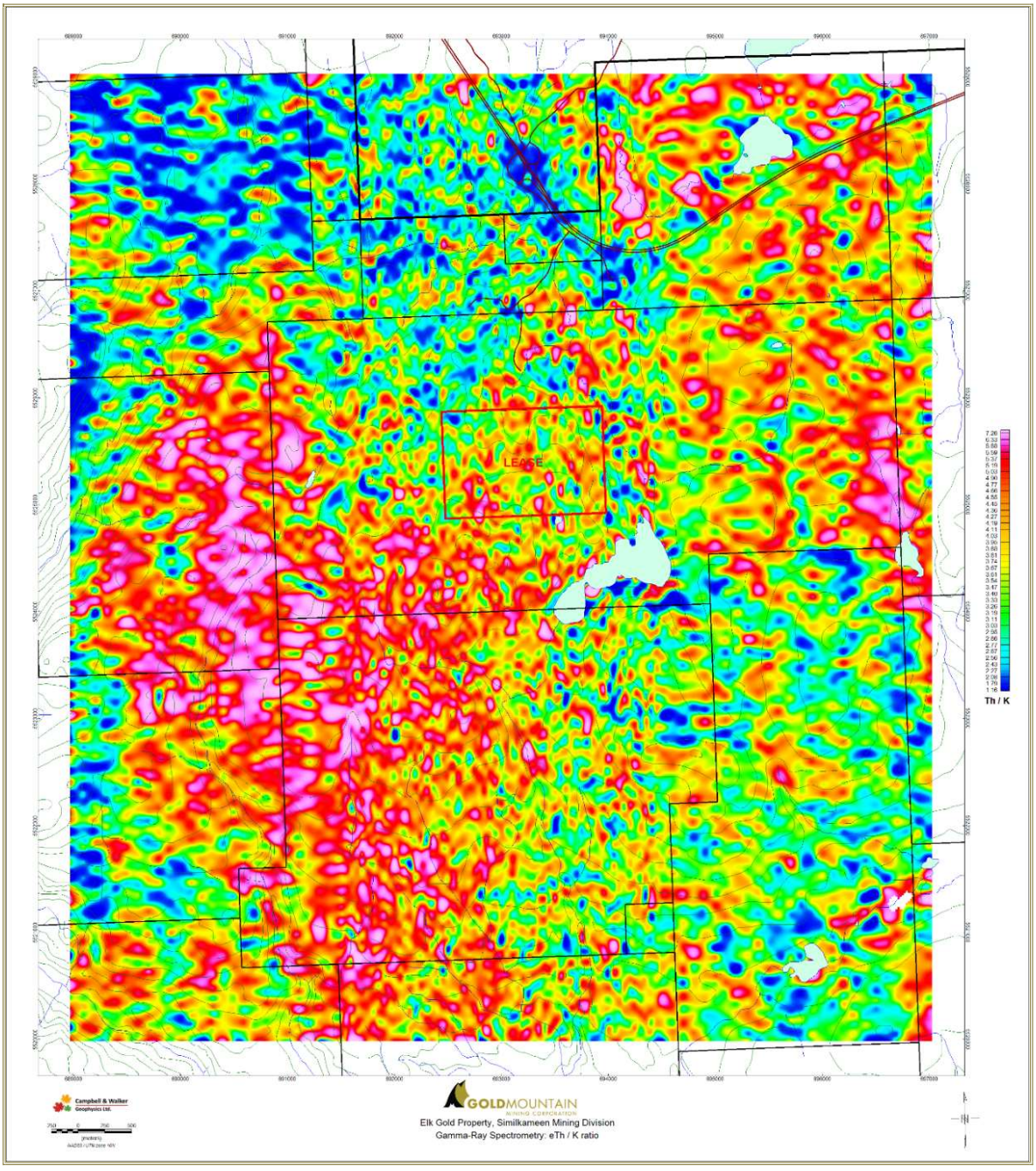
Anomalous entropy (e.g., 1 and 2 SD above the mean, shown as heavy black contour) delineates regions of significant magnetic discontinuity (i.e., geologic, including shear zones and lithological boundaries). If we accept that fault/shear zones play a role in the potential mineralization at the Elk Gold Project, then the irregular polygons mapped on the orientation entropy heat map should similarly reflect regions of significant structural complexity with high prospectivity for gold mineralization.

It is noteworthy that the mineralized zones containing known and drilled gold quartz veins lie within the warmest-coloured regions, e.g., Bullion, Siwash North, Lake, End, North (Discovery), and South Showings, as well as Siwash East along a northeast trend from Siwash North. The undrilled but anomalous soil geochemistry zone—Elusive—to the southwest also lies within one of the warm-coloured regions.

9.6 Gamma-Ray Spectrometry—Th/K ratio

Anomalous alteration associated with mineralization can often be distinguished from normal lithologic potassium variation by characteristic eTh/K ratio lows. Thorium enrichment generally does not accompany potassium during hydrothermal alteration processes, so eTh/K ratios provide excellent distinction between potassium associated with alteration and anomalies related to normal lithological variations. This important correlation of low eTh/K ratio with alteration is evident in countless studies worldwide.

High potassium levels are correlated with low Th/K ratios. The low Th/K anomaly defines mineralized structure; ratio anomalies are usually quite close in actual value to bedrock levels because factors such as vegetation cover and wetness, which would affect an individual element, affect both numerator and denominator and leave the ratio relatively unaffected. Unfortunately, characteristic low eTh/K ratio signatures do not appear to be correlated with any of the known mineralized showings on the Elk Gold Project (Figure 9-20). Further investigation is ongoing by the authors to establish possible reasons for this lack of markers.



Source: Campbell and Walker Geophysics Ltd., July 2019.

Figure 9-20: Low Th/K Areas of Interest

9.6.1 Discussion

The magnetic survey has provided valuable structural and geological information. A suite of filters and derivatives have been applied to the aeromagnetic gridded data, followed by a multi-scale edge detection, and automated grid "fractal" analysis to identify regions prospective for gold mineralization based upon structural complexity as inferred from magnetics being indicative of significant faults or shears. Known zones of mineralization correspond well with CET analysis maps, and it is notable that several untested zones also exist on the Property. Superimposing soil geochemical anomalies and known vein traces, together with follow-up ground study, are recommended for these regions.

Further information may also be obtainable from the current magnetic data through 3-D inversions, to identify discrete geometry of magnetic bodies. It is recommended that 3-D inversion modelling of the entire aeromagnetic dataset be conducted to identify contact zone dips.

10 DRILLING

From 1989 to 2013 nine different zones were tested by over 127,000 m of diamond, rotary, and percussion drilling in 1,237 holes. Initially, targets were drilled down dip of gold-mineralized quartz veins discovered by trenching soil geochemical anomalies. Those zones returning the best gold drill assays were targeted by additional drilling down dip and along strike of the intersections. Though the Siwash North zone was found from early drill campaigns to contain the best concentration of gold mineralization, several other zones that warranted drill testing and encountered gold-mineralized quartz veins are still open along strike and down dip.

10.1 Historical Drilling

Historical information in this section is largely sourced from assessment reports mainly written by Wojtek Jakubowski, P.Geo., and filed with BC's Ministry of Energy, Mines and Petroleum Resources. More recent drilling is documented within unpublished in-house reports (Alexander, 2011; Benoit, Saunders, & Wilson, 2013, 2014, and 2015; Benoit, Wilson, & Daubeny, 2012).

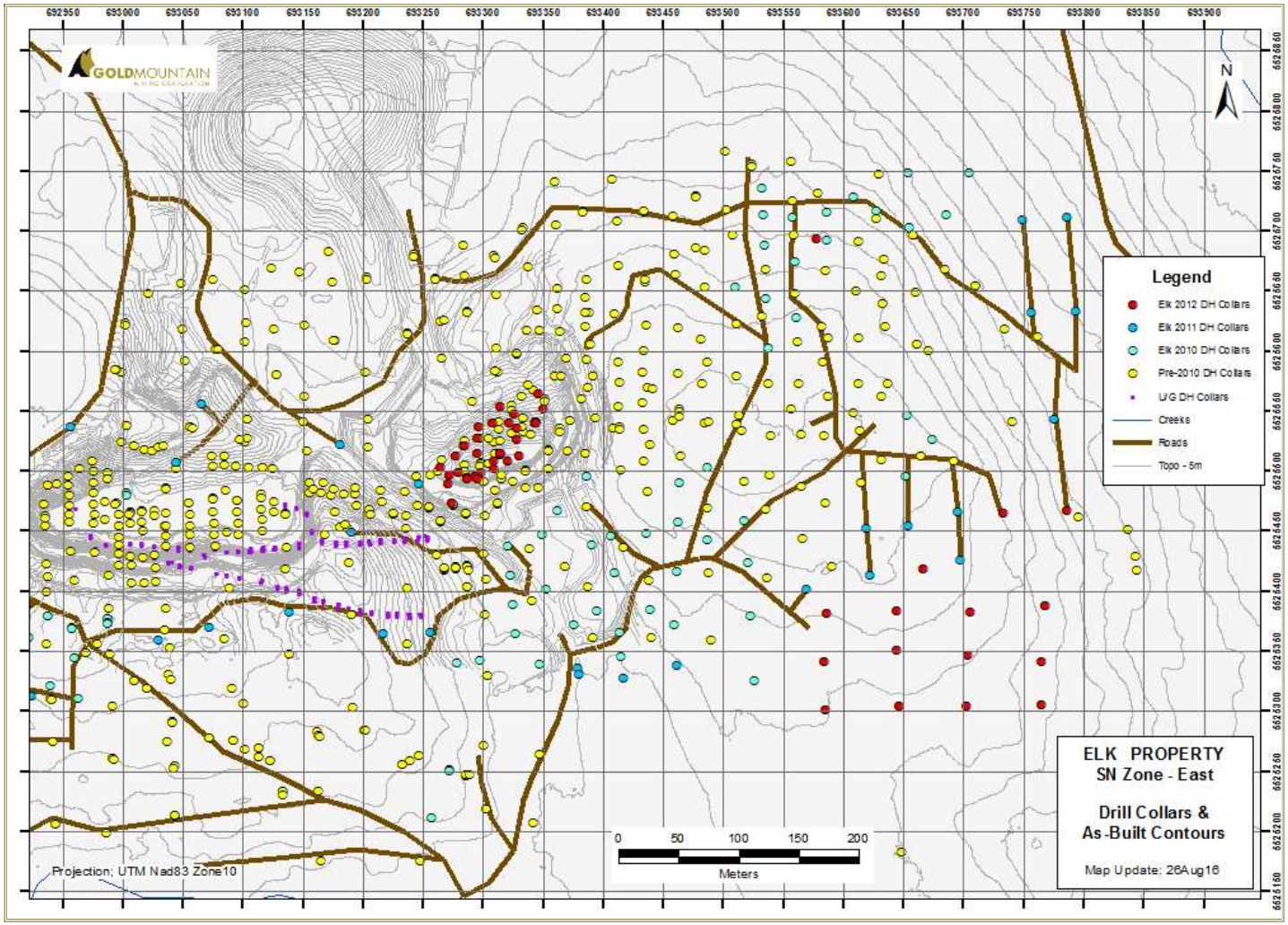
Diamond drilling on the Siwash North zone began in 1989, and in subsequent years also included drill testing of other target zones on the Property. Although gold-bearing quartz veins have been intersected in drill holes into these other zones, the drill spacing is not sufficient to calculate a compliant resource.

Drilling within the Siwash North zone may be viewed as occurring in three stages: 1) drilling by Cordilleran/Fairfield (1989–2001); 2) drilling by Almaden (2002–2010); and 3) drilling by GMMC (2011–2013). The current resource estimate includes all drilling to 2012. Figure 10-1 and Figure 10-2 display the drill collars for all historical and recent drilling . Since virtually all holes were drilled with a 360-degree azimuth the traces have been omitted for clarity of collar locations. Table 10-1 summarizes all diamond, rotary, and percussion drilling by zone and year.

Table 10-1: Diamond Drilling by Zone and Year

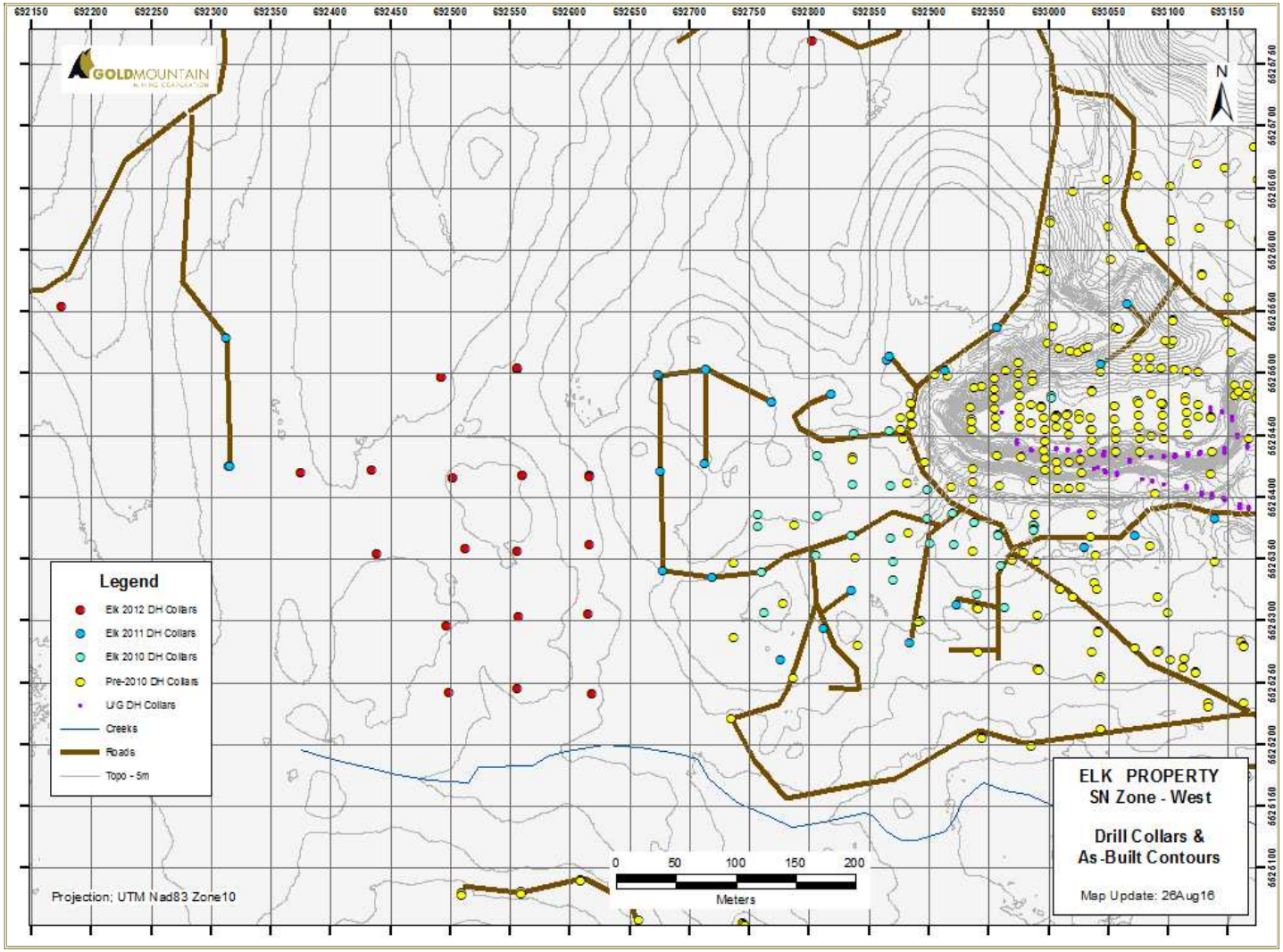
Zone	DH #	Meterage
Siwash North—Surface Diamond		
1989	12	778.9
1990	58	5,158.1
1991	37	6,587.0
1995	70	4,644.0
1996	88	6,947.9
2000	12	1,414.0
2002	26	5,006.2
2003	30	6,569.9
2004	44	10,298.5
2005	31	7,885.7
2006	54	8,368.4
2007	9	2,469.5
2010	87	12,749.2
2011	50	10,610.5
2012	84	9,259.0
Total	**692**	**98,746.8**
Siwash North—Underground Diamond		
1994	84	2,418.3
1995	217	7,600.3
Total	**301**	**10,018.6**
Siwash North—Reverse Circulation		
1992	90	3,626.0
Total	**90**	**3,626.0**
Siwash North—Percussion		
1992	19	257.0
2013	4	1,010.0
Total	**23**	**1,267.0**
Siwash East—Diamond		
2006	4	504.8
Total	**4**	**504.8**
Gold Creek—Diamond		
1996	7	556.9
2000	5	344.4
2002	4	333.1
2011	8	906.0
Total	**24**	**2,140.4**
Lake—Diamond		
1990	4	259.1
1995	7	477.0
2005	5	509.3
2011	8	1,125.0
2012	5	633.0
Total	**29**	**3,003.4**
Lake—Percussion		
2013	1	515.0
Total	**1**	**515.0**

Zone	DH #	Meterage
End/Great Wall—Diamond		
1995	6	289.0
2012	1	51.0
Total	7	**340.0**
Discovery—Diamond		
1995	6	397.0
2012	6	471.0
Total	12	**868.0**
South Showing—Diamond		
1995	9	480.97
2011	11	1,358.5
2012	20	2,622.6
Total	40	**4,462.07**
South Showing—Percussion		
2013	1	153
Total	1	**153**
Bullion Creek—Diamond		
2002	2	164.59
2004	4	394.64
2012	4	499
Total	10	**1,058.23**
Nicola—Diamond		
2012	3	366
Total	3	**366**
Total Diamond	1,122	**121,508.3**
Total Rotary & Percussion	115	**5561.0**
Grand Total	1,237	**127069.3**
Totals by Year		
1989	12	778.90
1990	62	5,417.20
1991	37	6,587.00
1992	109	3,883.00
1994	84	2,418.30
1995	315	13,888.27
1996	95	7,504.80
2000	17	1,758.40
2002	32	5,503.89
2003	30	6,569.90
2004	48	10,693.14
2005	36	8,395.00
2006	58	8,873.20
2007	9	2,469.50
2010	87	12,749.20
2011	77	14,000.00
2012	123	13,901.60
2013	6	1,678.00
Grand Total	1,237	**127,069.30**



Source: Bayshore, August 2020.

Figure 10-1: Siwash North Zone East—Historical and Recent Drill Collars



Source: Bayshore, August 2020.

Figure 10-2: Siwash North Zone West—Historical and Recent Drill Collars

10.2 Drilling Program (2010–2012)

10.2.1 Objectives

The 2010–2012 drilling program was initiated in October 2010, with the objectives of: 1) providing verification of the historical drill data, 2) testing down dip and along strike of known intersections to determine the potential to expand the resource base, and 3) providing additional drill-data density in areas of historical resources to assist in upgrading the classification during resource estimation.

10.2.2 Drilling Procedure

Drilling was carried out by multiple drill rigs belonging to different companies under the supervision of Brian Alexander, P.Geo., a contractor to Almaden (2010), Peter Thiersch, P.Geo., then Vice President of Exploration for GMMC (2011), and Robert G. Wilson, P.Geo., Senior Geologist for GMMC (2012). Drilling was done to industry standards with good-to-excellent core recovery.

Drilling was conducted 24 h/d in 12 h shifts, 7 d/wk. All drills were standard diesel-powered, hydraulic, wire-line coring machines. Drill pads were built using a small excavator, and the drill equipment was towed between sites by either a small bulldozer or skidder. Drill-hole collar locations were pre-marked in the field using a handheld GPS instrument. A series of back and fore sights were pre-marked before arrival of the drill rig to ensure proper drill alignment. Once on the pad, the rig's position was set using the sight flags. Drill return water and cuttings were collected in sumps which were refilled once dry. Drill collar locations were marked with a labelled PVC post after the drill moved off site, and later surveyed either by an independent survey company or by field staff using survey-grade RTK GPS units capable of sub-centimetre accuracies.

Control for historical drilling was by a mine grid oriented due north; recent drilling made use of GPS technology and employed UTM grid control. Due to a shift in datum elevations in 2012, all identifiable historical drill collars not destroyed by open pit mining and all recent drill collars were surveyed using the RTK GPS survey equipment. By comparing the RTK collar locations with those converted from mine grid to UTM coordinates, the 10.258 m vertical datum shift identified in Pooley et al. (2011) was confirmed. The re-survey was also able to confirm that original drill collar locations surveyed by T1 Transit, and tight metal chain was generally within 0.25 m of the RTK locations, though a few collars were out by up to 8 m. The current resource estimate update uses the re-survey drill collar coordinates.

Down-hole surveys for 2010–2012 drilling were made using single- and multi-shot REFLEX down-hole survey tools. A down-hole survey was usually taken within 25 m of surface (to confirm drill direction) and then approximately every 50 m afterwards.

Drill core was recovered in 10 ft long NQ or HQ core tubes and placed in 4 ft or 5 ft wooden core boxes with wooden blocks marking the footage. The core was normally delivered to the core shack on the Property at the end of each drill shift. Upon receipt of the core the staff geotechnical crew converted hole depth tags in each core box to metric measurements and top and bottom hole depths for each core box

were similarly converted to metric. All subsequent core logging or measurements were conducted in metric. Core was washed and a geotechnical log was completed, recording box top and bottom measurement, the percent recovery, RQD, number of fractures, and number of low-angle fractures. Metal box tags were inscribed with the hole number and box from–to details and affixed to the box fronts. Geologists then logged the core to a laptop computer using a modified free-form log allowing collection of both descriptive and numerical data recording. Geologists recorded information about lithology, alteration, mineralization, and structure, as well as sample location details. All drill-core was photographed in sets of four boxes per frame, following marking of sample locations but prior to sawing or splitting.

All gold-bearing quartz veins on the Elk Gold Project dip shallowly to moderately to the south, hence drill holes are drilled to the north at angles between −45° and −85° depending on the desired depth of vein intersection. Though the veins have minor changes in strike direction along their length, the changes generally even out such that vein intersections can be predicted with down-hole accuracies of less than 3 m.

10.2.3 Drill Results

Within the Siwash North zone, drilling in 2010, 2011, and 2012 was completed mainly on three areas: the Siwash North western extension, the Siwash North southeast (down-dip) extension, and the bulk sample Pit 2 area (Table 10-2). The 2010 drill program focused on 1000 series veins immediately west and down dip of Pit 1, as well as down dip and along strike east of historical drilling. Additional holes in 2011 sought to confirm grades and widths of 2000 series veins, and test down-dip extensions of known 1000 vein mineralization. In 2012 Siwash North western extension drilling sought to link known 1300 vein mineralization from the historical pit area west towards the formerly named Far West zone. The South East Extension drilling examined the down-dip extent of the 1300 vein east of both the bulk sample it and the area tested in 2010. Drilling in the bulk sample Pit 2 area was designed to provide, in advance of mining, detailed data about the nature and grade of the 1000 series veins.

Table 10-2: Summary of Siwash North Drill Programs 2010–2012

Year	Number of Drill Holes	Total Metres
2010	87	12,749.2
2011	50	10,610.5
2012	84	9,259.0
Total	221	32,618.7

10.2.3.1 Drill Results 2010

Drilling in 2010 was completed under Almaden corporate direction, with Brian Alexander, P.Geo., supervising the work. Information in this section is taken from Alexander (2011) and from published Almaden news releases. Within the Siwash North Zone 87, drill holes were completed (Figure 10-3) in three main areas: west of Pit 1, testing the 1000 series veins along strike and down dip as well as infill holes; southeast of Pit 1, testing 1000 series veins and adding infill to historical drilling; and northeast of Pit 1 to add infill drilling to the 2000 series veins.

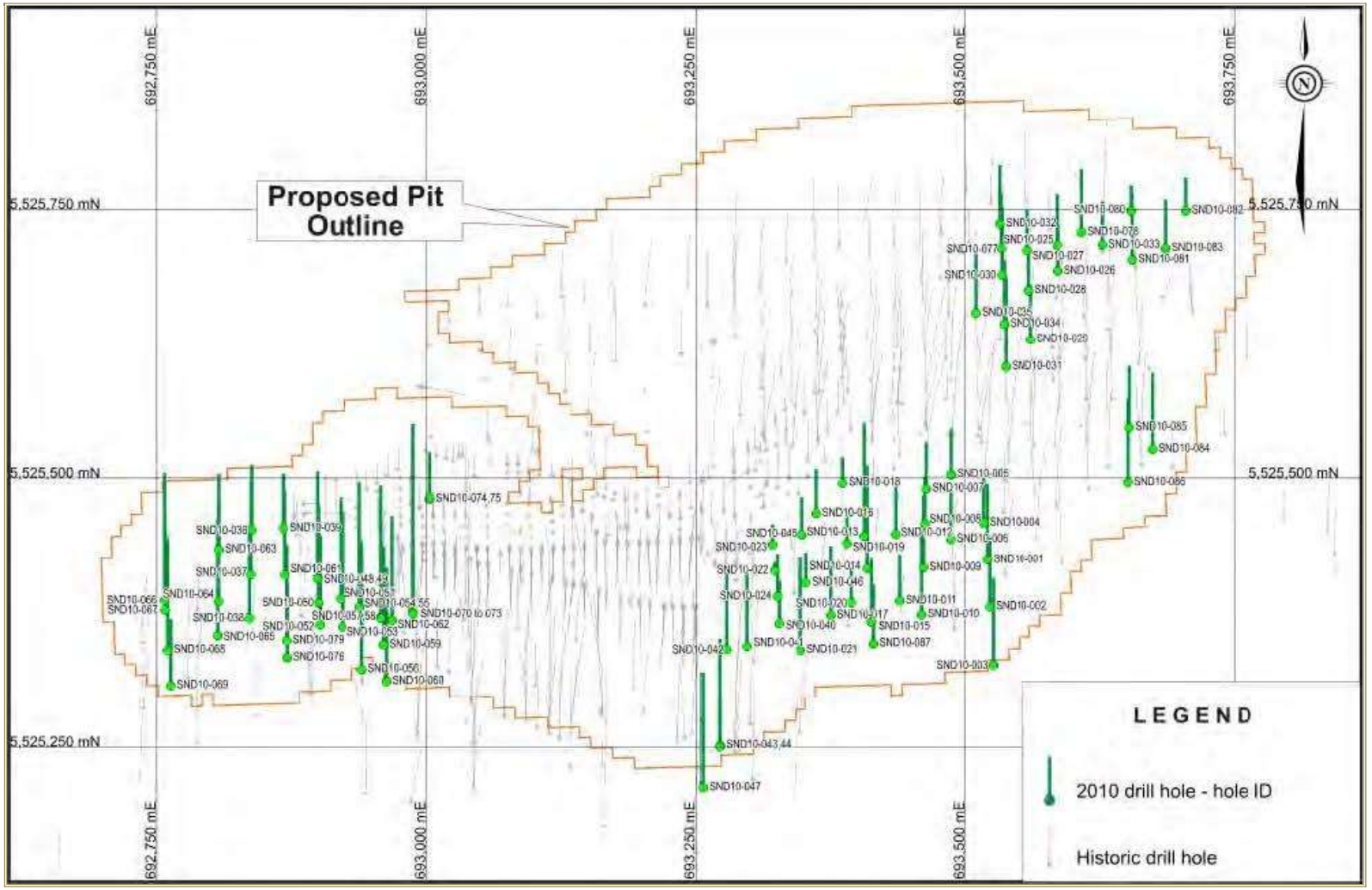
Results from the 2010 drill program are shown in Table 10-3 to Table 10-7 and were drawn from Almaden 2011 news releases that also commented on the assaying and composites. Each Elk Gold Project news release from that year contained the following disclaimer:

The analyses reported were carried out at ALS Chemex Laboratories of North Vancouver using industry standard aqua regia, ICP and fire assay techniques. Blanks, field duplicates and certified standards were inserted into the sample stream as part of Almaden's quality assurance and control program which complies with National Instrument 43-101 requirements. Intervals that returned assays below detection were assigned zero values. Composites were calculated using a cut-off grade of 0.25 g/t gold and with a maximum of 5 meters internal dilution (values below 0.25 g/t). Reported widths are intersection and not true widths (Almaden, 2011).

Table 10-3 also contains results from South Showing drilling that is not part of the Siwash North resource estimate.

10.2.3.2 Drill Results 2011

Drilling in 2011 was completed under the corporate direction of GMMC, with Peter Thiersch, P.Geo., supervising the work. The strategy for the 2011 drilling was to circle the deposit with step-out drilling, particularly along the east and west strike and dip extensions, with some minor infill drilling in the central deposit area, Figure 10-4. Significant intersections from that drilling are presented in Table 10-7.



Source: Bayshore, August 2020.

Figure 10-3: 2010 Drill Collars (Pit Outline No Longer Valid)

Table 10-3: Significant Results from Holes SND10-001 to SND10-020

Zone	Hole #	From (m)	To (m)	Width (m)	Gold (g/t)
South	SSD 10-003	46.50	55.79	9.29	0.65
South	SSD 10-003	11.80	15.00	3.20	1.14
South	SSD 10-004	39.00	46.81	7.81	1.04
Resource Area	SND 10-001	13.60	14.00	0.40	4.14
Resource Area	SND 10-002	33.00	34.34	1.34	1.12
Resource Area	SND 10-004	106.70	108.45	1.75	5.35
Resource Area	SND 10-005	63.85	65.30	1.45	3.21
Resource Area	SND 10-005	87.80	89.60	1.80	6.76
Resource Area	SND 10-006	98.30	100.70	2.40	14.87
Resource Area	SND 10-007	79.30	85.50	6.20	0.56
Resource Area	SND 10-008	71.95	74.50	2.55	2.44
Resource Area	SND 10-009	92.70	96.30	3.60	1.99
Resource Area	SND 10-010	94.25	105.45	11.20	0.31
Resource Area	SND 10-010	146.10	148.30	2.20	13.08
Resource Area	SND 10-011	83.03	89.55	6.52	23.74
Resource Area	including	86.30	89.55	3.25	46.70
Resource Area	SND 10-012	71.35	83.80	12.45	2.97
Resource Area	including	71.35	72.40	1.05	9.31
Resource Area	and	78.68	83.80	5.12	5.14
Resource Area	and	78.68	80.40	1.72	15.12
Resource Area	SND 10-013	56.95	66.80	9.85	1.31
Resource Area	SND 10-013	72.20	84.60	12.40	1.51
Resource Area	including	72.20	76.00	3.80	5.16
Resource Area	SND 10-014	61.00	67.25	6.25	0.73
Resource Area	SND 10-014	71.95	76.25	4.30	1.73
Resource Area	SND 10-015	78.95	83.15	4.20	1.33
Resource Area	SND 10-015	99.77	103.25	3.48	0.81
Resource Area	SND 10-016	30.70	38.20	7.50	0.39
Resource Area	SND 10-017	94.00	95.50	1.50	26.40
Resource Area	SND 10-018	31.15	61.00	29.85	1.46
Resource Area	including	40.65	42.05	1.40	11.26
Resource Area	and	50.85	52.35	1.50	12.61
Resource Area	SND 10-019	43.45	54.75	11.30	0.40
Resource Area	SND 10-019	64.20	68.20	4.00	0.52
Resource Area	SND 10-020	68.00	68.30	0.30	13.75
Resource Area	SND 10-020	100.90	104.05	3.15	1.26

Table 10-4: Significant Results from Holes SND10-021 to SND10-048

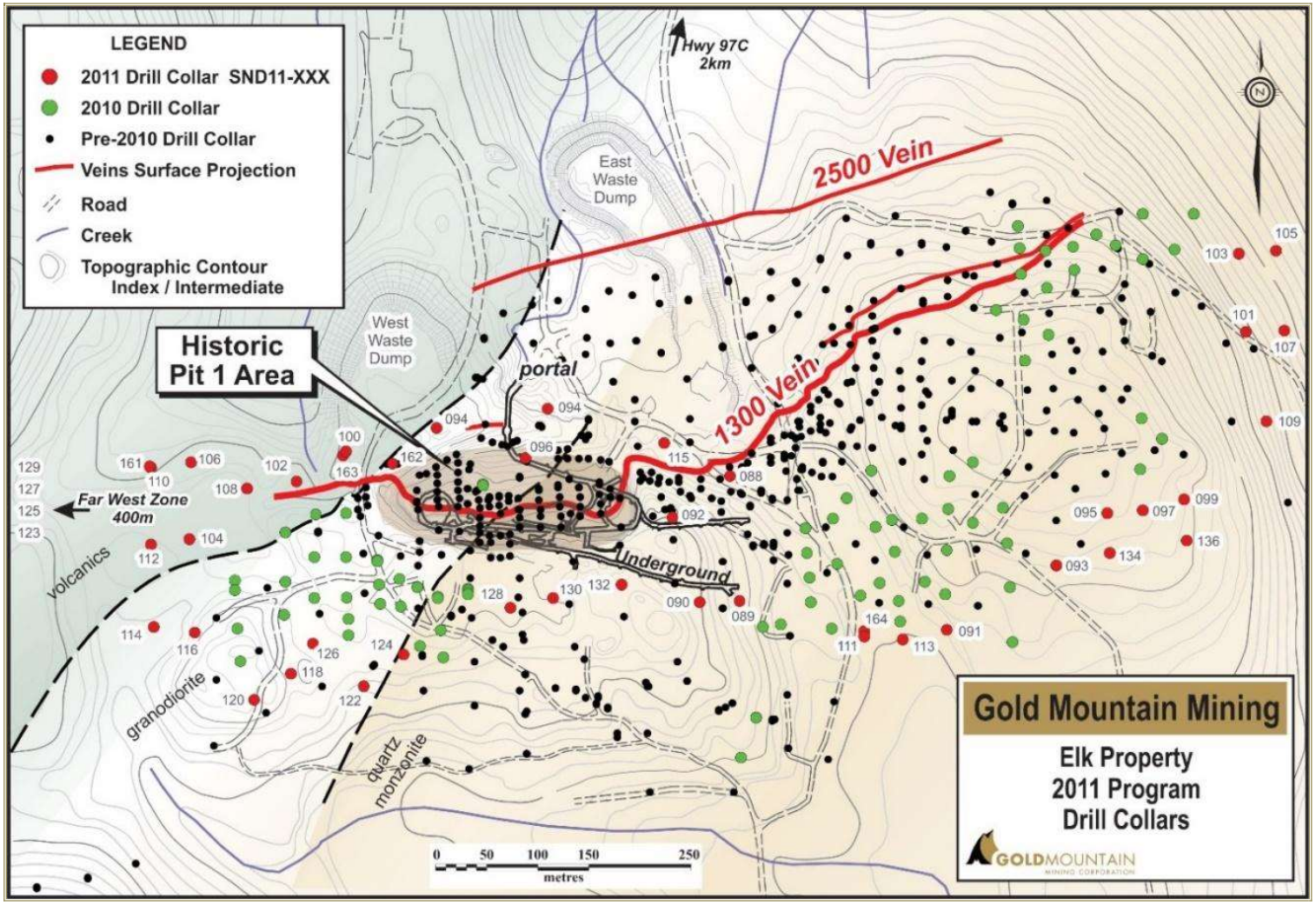
Zone	Hole #	From (m)	To (m)	Width (m)	Gold (g/t)
Resource Area	SND10-21	140.80	141.90	1.10	2.23
Resource Area	SND10-22	116.45	118.38	1.93	0.54
Resource Area	SND10-23	33.50	33.85	0.35	0.54
Resource Area	SND10-24	100.47	101.00	0.53	10.45
Resource Area	SND10-24	106.95	107.70	0.75	1.41
Resource Area	SND10-25	91.42	91.66	0.24	8.32
Resource Area	SND10-26	88.40	88.90	0.50	11.09
Resource Area	SND10-27	63.20	68.10	4.90	1.29
Resource Area	SND10-27	89.40	90.70	1.30	4.45
Resource Area	SND10-28	121.70	123.70	2.00	1.39
Resource Area	SND10-29	175.86	178.00	2.14	6.28
Resource Area	SND10-32	20.37	20.65	0.28	62.80
Resource Area	SND10-32	54.20	55.70	1.50	0.53
Resource Area	SND10-32	82.10	85.65	3.55	0.62
Resource Area	SND10-34	24.65	28.00	3.35	2.87
Resource Area	SND10-34	32.80	44.85	12.05	0.51
Resource Area	SND10-40	116.45	118.38	1.93	0.54
Resource Area	SND10-41	98.95	101.10	2.15	0.65
Resource Area	SND10-41	111.00	118.35	7.35	0.37
Resource Area	SND10-41	149.20	150.20	1.00	7.07
Resource Area	SND10-42	101.00	115.50	14.50	1.96
Resource Area	including	101.00	102.80	1.80	14.57
Resource Area	SND10-43	47.50	47.60	0.10	13.35
Resource Area	SND10-44	182.15	182.90	0.75	11.93
Resource Area	SND10-45	21.60	23.45	1.85	1.66
Resource Area	SND10-45	59.10	63.17	4.07	1.33
Resource Area	SND10-46	36.27	36.50	0.23	1.76
Resource Area	SND10-47	27.90	28.10	0.20	141.50
Resource Area	SND10-47	198.40	199.90	1.50	5.46
Resource Area	SND10-48	148.80	150.20	1.40	3.20

Table 10-5: Significant Results from Holes SND10-049 to SND10-065 Including Additional Assays from SND10-30 to 39

Zone	Hole #	From (m)	To (m)	Width (m)	Gold (g/t)
Resource Area	SND-10-30	13.16	13.40	0.24	3.81
Resource Area	SND-10-30	41.70	41.90	0.20	4.19
Resource Area	SND-10-30	97.70	101.75	4.05	0.70
Resource Area	includes	28.86	29.84	0.98	1.19
Resource Area	SND-10-31	41.20	41.45	0.25	15.55
Resource Area	SND-10-31	66.15	68.68	2.53	2.87
Resource Area	SND-10-31	184.93	186.35	1.42	6.95
Resource Area	SND-10-33	17.27	17.59	0.32	9.12
Resource Area	SND-10-34	24.65	28.00	3.35	2.87
Resource Area	SND-10-34	32.80	44.85	12.05	0.51
Resource Area	SND-10-34	152.00	156.35	4.35	6.96
Resource Area	includes	152.00	153.10	1.10	26.56
Resource Area	SND-10-35	7.94	16.90	8.96	0.72
Resource Area	includes	7.94	8.17	0.23	18.90
Resource Area	SND-10-37	129.25	132.35	3.10	1.78
Resource Area	SND-10-38	128.84	129.12	0.28	4.64
Resource Area	SND-10-38	153.85	154.78	0.93	0.92
Resource Area	SND-10-39	107.05	111.37	4.32	3.86
Resource Area	includes	109.75	110.10	0.35	45.70
Resource Area	SND-10-50	187.80	190.50	2.70	3.50
Resource Area	includes	189.30	190.50	1.20	8.62
Resource Area	SND-10-51	28.96	29.22	0.26	4.52
Resource Area	SND-10-51	97.20	98.00	0.80	0.71
Resource Area	SND-10-52	213.00	215.55	2.55	2.58
Resource Area	SND-10-54	85.75	87.30	1.55	1.00
Resource Area	SND-10-55	105.75	110.85	5.10	0.75
Resource Area	SND-10-56	191.46	192.32	0.86	0.63
Resource Area	SND-10-57	100.30	100.70	0.40	1.62
Resource Area	SND-10-59	53.03	54.55	1.52	3.30
Resource Area	SND-10-59	146.58	153.80	7.22	2.39
Resource Area	includes	153.00	153.80	0.80	21.00
Resource Area	SND-10-60	208.70	209.00	0.30	13.95
Resource Area	SND-10-61	70.75	71.93	1.18	0.63
Resource Area	SND-10-61	144.47	146.15	1.68	0.46
Resource Area	SND-10-62	110.05	112.00	1.95	3.13
Resource Area	SND-10-62	177.42	177.95	0.53	11.71
Resource Area	SND-10-64	80.95	83.40	2.45	15.36
Resource Area	includes	82.10	83.40	1.30	28.23
Resource Area	SND-10-65	114.60	114.91	0.31	17.80

Table 10-6: Significant Results from Holes SND10-065 to SND10-087 Including Additional Assays from SND10-30 to 39

Zone	Hole #	From (m)	To (m)	Width (m)	Gold (g/t)
Resource Area	SND-10-65	152.14	153.00	0.86	2.53
Resource Area	SND-10-67	76.50	82.70	6.20	0.44
Resource Area	SND-10-67	162.88	164.26	1.38	0.89
Resource Area	SND-10-68	121.05	129.15	8.10	0.35
Resource Area	SND-10-68	201.84	207.78	5.94	0.27
Resource Area	SND-10-69	34.77	35.35	0.58	5.73
Resource Area	SND-10-69	165.05	165.61	0.56	0.28
Resource Area	SND-10-70	29.22	32.98	3.76	1.99
Resource Area	SND-10-71	98.05	99.77	1.72	4.40
Resource Area	SND-10-73	38.95	39.30	0.35	4.27
Resource Area	SND-10-74	8.61	9.47	0.86	1.67
Resource Area	SND-10-74	39.99	40.84	0.85	3.08
Resource Area	SND-10-77	47.27	48.77	1.50	7.58
Resource Area	includes	48.27	48.57	0.30	35.90
Resource Area	SND-10-77	74.74	76.41	1.67	0.95
Resource Area	includes	76.15	76.41	0.26	5.74
Resource Area	SND-10-78	86.91	89.21	2.30	1.33
Resource Area	SND-10-79	134.70	135.20	0.50	14.25
Resource Area	SND-10-79	158.50	159.30	0.80	25.70
Resource Area	SND-10-79	195.15	197.50	2.35	5.10
Resource Area	SND-10-80	20.61	21.70	1.09	5.93
Resource Area	SND-10-83	63.30	68.00	4.70	0.89
Resource Area	includes	63.30	64.60	1.30	2.66
Resource Area	SND-10-84	56.02	57.33	1.31	0.71
Resource Area	SND-10-84	92.73	93.68	0.95	3.77
Resource Area	SND-10-84	102.34	103.82	1.48	0.61
Resource Area	SND-10-85	33.86	34.88	1.02	0.81
Resource Area	SND-10-86	39.12	40.58	1.46	1.14
Resource Area	SND-10-86	70.01	74.20	4.19	1.44
Resource Area	includes	70.01	70.32	0.31	10.15
Resource Area	includes	73.79	74.20	0.41	6.33
Resource Area	SND-10-86	116.27	117.05	0.78	1.30
Resource Area	SND-10-87	65.20	85.10	19.90	1.38
Resource Area	includes	65.20	65.40	0.20	14.65
Resource Area	includes	67.50	74.00	6.50	2.51
Resource Area	SND-10-87	78.00	83.00	5.00	1.43



Source: Bayshore, August 2020.

Figure 10-4: 2011 Drill Collars

Table 10-7: Significant Intersections from 2011 Drilling in Siwash North Zone

Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)
SND11-088	281.82	283.60	**1.78**	**22.5**	**17.0**
Including	*281.82*	*282.08*	*0.26*	*154.0*	*113.0*
SND11-090	91.75	93.15	**1.40**	**19.2**	**21.5**
Including	*91.75*	*91.97*	*0.22*	*105.0*	*104.0*
And	396.95	398.64	**1.69**	**15.0**	**17.6**
Including	*398.08*	*398.34*	*0.26*	*96.0*	*108.0*
SND11-092	34.40	35.80	**1.40**	**25.3**	**13.8**
Including	*35.40*	*35.60*	*0.20*	*168.0*	*42.2*
SND11-096	136.45	138.12	**1.67**	**4.76**	**8.5**
Including	*137.20*	*137.62*	*0.42*	*18.9*	*33.8*
SND11-105	167.42	168.62	**1.20**	**4.45**	**210.6**
Including	*167.42*	*167.92*	*0.50*	*10.6*	*500.0*
SND11-106	143.50	144.76	**1.26**	**4.73**	**35.7**
Including	*144.20*	*144.45*	*0.25*	*23.4*	*168.0*
SND11-110	128.10	129.56	**1.46**	**16.6**	**141.5**
Including	*128.80*	*129.00*	*0.20*	*115.0*	*860.0*
SND11-112	41.21	42.45	**1.24**	**22.3**	**54.9**
Including	*41.41*	*41.61*	*0.20*	*126.0*	*286.0*
SND11-113	86.70	88.33	**1.63**	**14.2**	**8.5**
Including	*87.41*	*87.62*	*0.21*	*109.0*	*64.0*
SND11-115	229.42	230.83	**1.41**	**9.13**	**5.8**
Including	*229.77*	*229.97*	*0.20*	*64.0*	*40.2*
SND11-123	73.67	74.75	**1.08**	**10.9**	**20.7**
Including	*74.13*	*74.52*	*0.39*	*30.4*	*56.2*
SND11-128	131.10	132.7	**1.60**	**159.9**	**118.3**
Including	*132.00*	*132.55*	*0.55*	*465.0*	*344.0*
SND11-132	292.10	293.30	**1.20**	**30.33**	**95.8**
Including	*292.60*	*292.80*	*0.20*	*182.0*	*575.0*

The following summary of the 2011 drilling is from Benoit, Wilson, and Daubeny (2012).

Drilling in 2011 northwest of Pit 1 added about 300 m westerly extension to the previously known 2000 vein set, with good results received in holes drilled under and west of the west waste dump. Two veins were intersected in this area, both somewhat flatter and below/north of the steeper 2500 vein, and are thought to represent extensions of the 2600 and 2700 veins.

Drilling southwest of the Pit 1 area, along the southern/depth extension of the 1000 series veins produced generally poor results and this area does not warrant additional follow-up at depth.

Drilling southeast of the deposit returned mixed results. Targets were the southern/depth extension of the flatter 1000 series veins east of the RBF. Some high-grade intersections in this area may still warrant infill follow up.

Drilling in the northeast deposit area was directed at extending the 2000 series veins to the east, and returned mixed results. Two holes were lost due to bad ground conditions and grades were generally low, perhaps the result of a large northwest-trending cross fault which is inferred to exist in the Siwash Creek drainage immediately to the east. Nevertheless, several veins were intersected in the footwall to the 2500 vein, suggesting good up-dip potential for discovery of new veins north of the 2500 vein. This has essentially been confirmed with the discovery of the 2600 and 2700 veins and bodes well for additional exploration north of the Pit 1 towards the Bullion zone target.

In the central deposit area several long holes were directed to test both the 1300 and 2500 veins, and the large area between these two main structures. Results were similar to historical results, in that many small veins occur sporadically between these two structures, but grades are generally very low. It appears that the highest and most consistent grades are restricted to the 1300 and 2500 veins and flatter 1000 series veins east of the RBF. All veins remain open to depth.

10.2.3.3 Drill Results 2012

Drilling in 2012 was completed under the corporate direction of GMMC, with Robert G. Wilson, P.Geo., supervising the work. Within the Siwash North zone, 50 diamond-drill holes were drilled for 4,430 m of drilling. The drill program in this zone was designed to test three areas: the 1000 series veins west of Pit 1, and into the Nicola Group volcanic rocks where a gold-mineralized quartz vein was intersected in 2011 drilling; an area along the southeast of historical drilling to test the down-dip extent of 1000 series veins; and infill drilling in the then-proposed Pit 2 area to provide control for planned bulk sample mining (Figure 10-1 and Figure 10-2). Significant intersections from that drilling are presented in Table 10-8. The following summary of that program is drawn from Benoit and Wilson (2013).

Drill targets in the Siwash North zone consist of two sets of sub-parallel auriferous quartz veins of varying widths and grades hosted within Osprey Lake quartz monzonite, granodiorite, and diorite, as well as Nicola Volcanic rocks to the west. The two vein sets are named the 1000 series and the 2000 series, with individual veins within the sets denoted using intervals of 100 based on their locations in the series (e.g., 1300 vein is the third major vein of the 1000 series). The 1000 series has been separated into eight unique veins and the 2000 series into six unique veins. The main distinguishing factor in separating a 2000 series vein from a 1000 series vein is that the 2000 series veins generally have a steeper dip than the 1000 series veins. The 1000 series veins typically strike 090 (right-hand rule—RHR) and dip to the south between 15° and 30°. The exception to this is the 1300 vein, which changes its orientation across the deposit. On the west end the 1300 vein dips steeply at 60° while in the centre of the known strike length it changes dip with depth increasing from 20° to 60° at an approximate elevation of 1,560 masl. On the east end of the Property the 1300 vein dips shallowly at 15° to 25°. The 2000 series veins come to surface 200 m north of the 1000 series vein trenches, generally have steeper dips ranging from 40° to 60°, with strikes ranging

from 060 to 085 (RHR). These strikes and dips are all based on the working solid/wireframe model. The 1300 vein was mined from 1992 to 1995 and produced over 50,000 oz of gold, and has been the primary area of interest in recent exploration programs.

Table 10-8: Significant Intersections from 2012 Drilling in Siwash North Zone

Hole	Area	Width (m)	Au Grade (g/t)	Depth (m)	Grade x Width	Comments
SND12-184	Far West Extension	0.33	16	41.2	4.80	Composites to 6.85 g/t Au and 27.11 g/t Ag over 0.8 m width, or to 1.33 g/t Au and 19.61 g/t Ag over 3.1 m width
SND12-186	Far West Extension	0.4	19.85	158.6	7.94	Composite to 7.03 g/t Au over 1.2 m width
SND12-256	Far West Extension	0.3	21.5	128.15	6.45	2 Duplicates average, composite to 4.47 g/t Au and 14.46 g/t Ag over 1.6 m width
SND12-166	Bulk Sample Pit	0.62	63.1	25.36	39.12	Composite to 15.02 g/t Au over 2.89 m width
SND12-169	Bulk Sample Pit	0.9	35.1	23.73	31.59	Composite to 13.7 g/t Au over 2.47 m width
SND12-173	Bulk Sample Pit	0.25	24.1	19.25	6.03	Composites to 7.55 g/t Au over 1.96 m width, or 4.29 g/t Au over 3.98 m width
SND12-174	Bulk Sample Pit	0.3	20	17.73	6.00	36.3 g/t Ag
SND12-193	Bulk Sample Pit	0.7	18.15	11.3	12.71	48.9 g/t Ag
SND12-194	Bulk Sample Pit	0.5	16.9	14	8.45	50.5 g/t Ag
SND12-195	Bulk Sample Pit	0.4	19.65	3.3	7.86	11.2 g/t Ag
SND12-196	Bulk Sample Pit	0.5	16.9	18.47	8.45	23.5 g/t Ag
SND12-197	Bulk Sample Pit	0.5	88.8	21.5	44.40	121 g/t Ag
SND12-197	Bulk Sample Pit	0.2	19.85	10.95	3.97	1200 vein
SND12-199	Bulk Sample Pit	0.74	30.2	28.86	22.35	Composite to 7.5 g/t Au over 3.59 m width
SND12-199	Bulk Sample Pit	0.5	25.2	15.5	12.60	Composite to 10.47 g/t Au over 1.5 m width
SND12-200	Bulk Sample Pit	0.5	49.45	9.25	24.73	2 Duplicates average, 64.45 g/t Ag average
SND12-214	Bulk Sample Pit	0.5	17.65	13.37	8.83	47.3 g/t Ag
SND12-276	Bulk Sample Pit	0.5	26.5	11.5	13.25	Composite to 20.75 g/t Au over 1 m width
SND12-268	SE Extension	0.3	16.8	132.2	5.04	10.2 g/t Ag
SND12-275	SE Extension	0.31	23.9	179.1	7.41	24.2 g/t Ag

The Siwash North zone was drill tested in three areas: The Far-West Extension, the bulk sample area, and the South-East Extension. Within these three areas 9,259 m of drilling was completed in 84 drill holes.

Far-West Extension

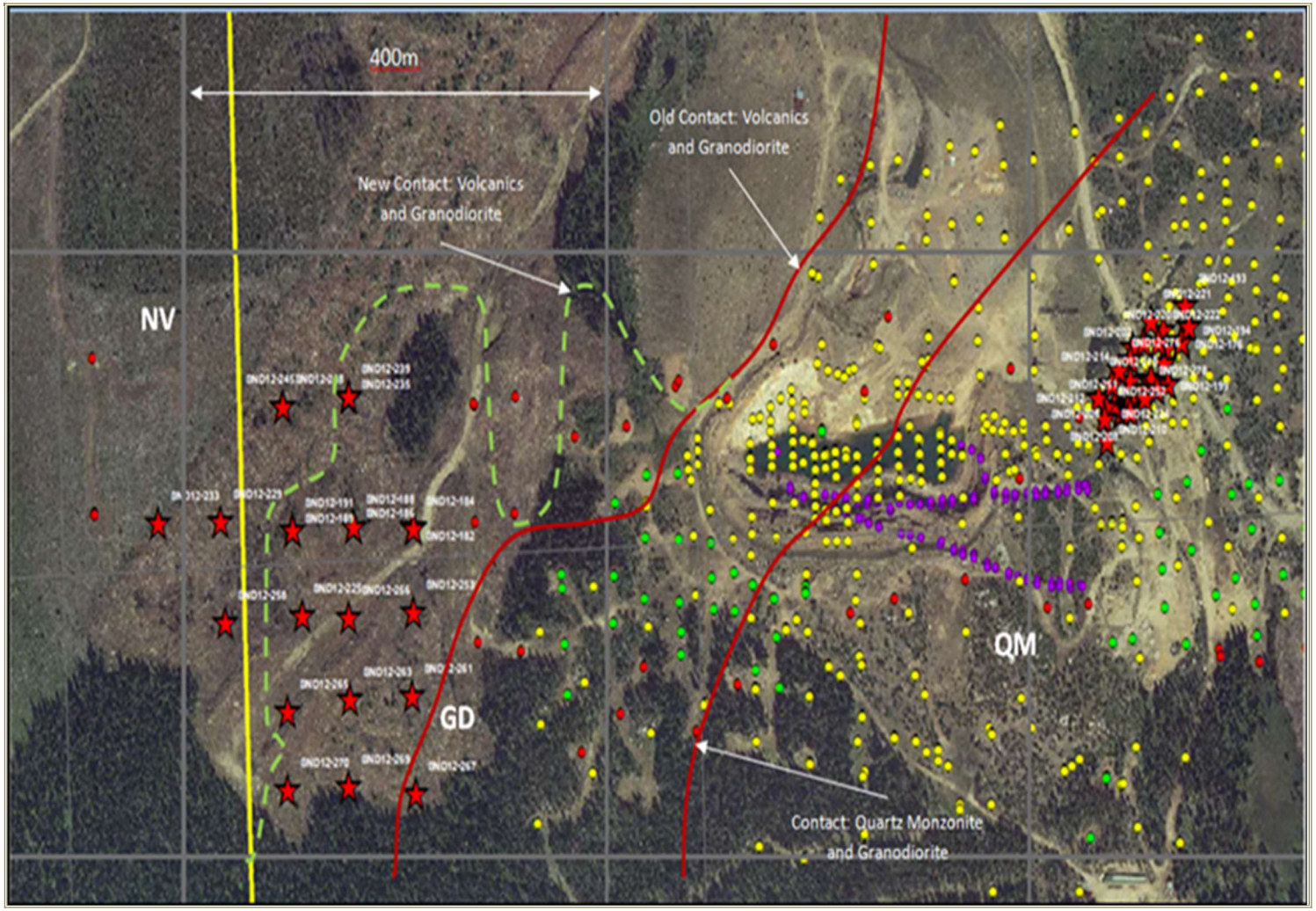
In 2012 the 1000 series vein system was grid drilled between the western end of the historical pit and a 400 m step-out hole drilled in 2011 (SND11-123) that encountered a gold-bearing quartz vein hosted in Nicola volcanic rocks. Twenty-two holes (3,245 m) were drilled to extend the Siwash North resource westward and to verify the continuity and grade of 1000 series veins into Nicola Group rocks. This area was previously known as the Far West zone and has now been linked with the Siwash North zone.

Thirty-seven holes (1,507.5 m) were drilled in the bulk sample Pit 2 area to provide control for bulk sample mining that started in September 2012. On the southeastern end of the Siwash North zone (Southeast Extension), an additional fifteen holes (3,024.5 m) were drilled to follow the 1000 series veins east along strike and down dip.

The Far-West Extension is now considered to be linked with the Siwash North zone based on the 2012 results, confirming the 400 m western step-out between the historical pit western end and hole SND11-123 where a 30 cm quartz vein grading 30.4 g/t Au and 56 g/t Ag was intersected within an intermediate volcanic host. Holes drilled in 2012 had a north-facing azimuth to intersect the south-dipping 1000 series vein structures. A grid-pattern drilling program, with a drill-collar spacing of 40 m to 60 m was employed to delineate this possible western strike extension. Previous drill exploration in the area was limited and therefore geological contacts were based on outcrop observations, IP results, as well as sparse magnetometer data.

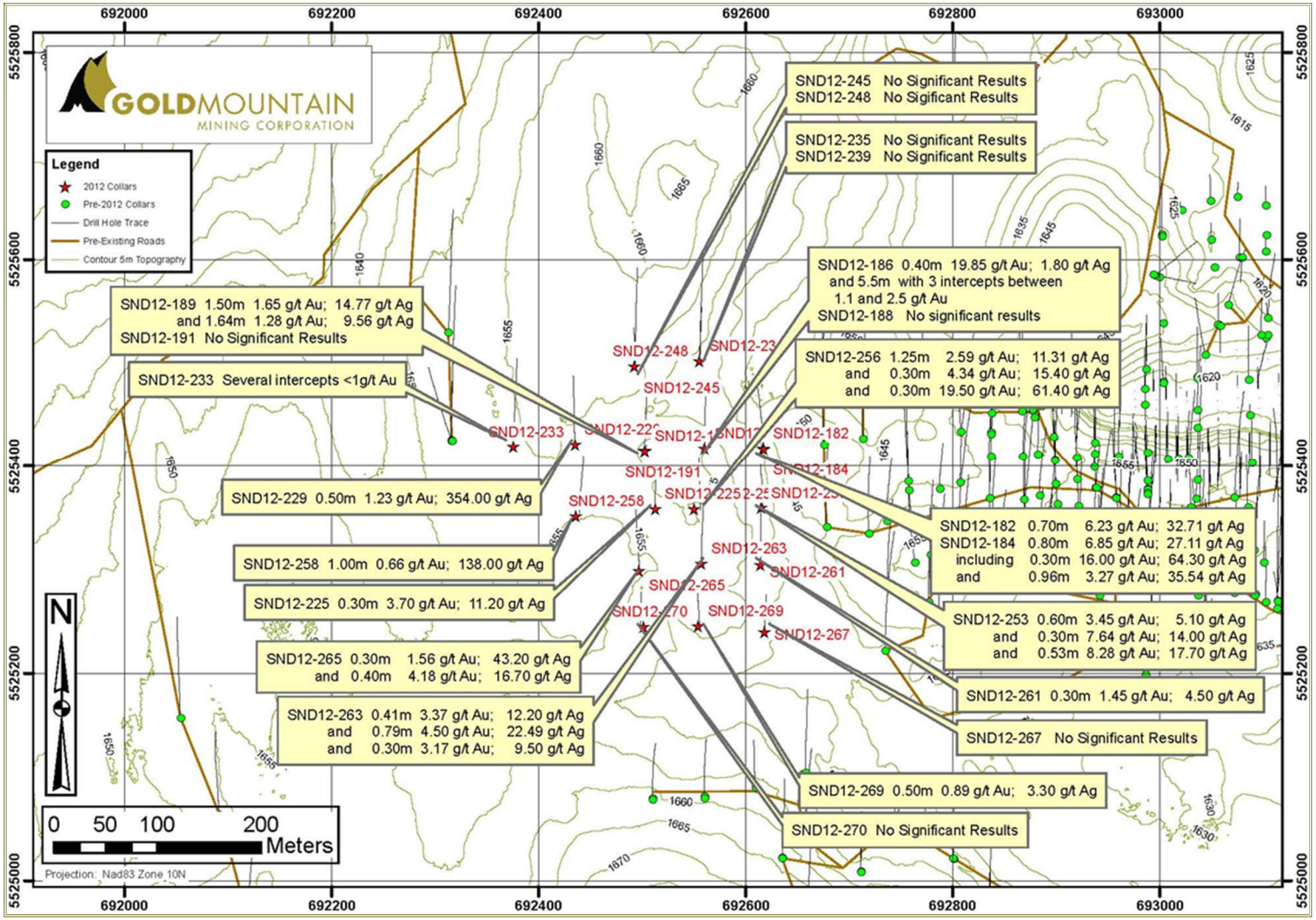
The 2012 drill program helped to better delineate the contacts between the Nicola and Osprey Lake Group rocks (Figure 10-5). Historically, it was thought that mineralization was restricted to only the diorite, granodiorite, and quartz monzonite. The drilling shows that quartz veining containing pyrite as well as base metals and gold does occur in the Nicola, but the volcanic rocks do not support significant vein development within them. Narrower and less continuous veins in the Nicola may be due to the volcanic rocks being less brittle than the intrusive units that dilate easier and remain open during gold-enriched quartz emplacement.

Although Far-West Extension drilling in 2012 showed that a gold-bearing quartz vein system is present, results are not consistently high enough grade to warrant additional work, and no recommendations are made for follow-up drilling. Significant results are shown in Figure 10-6.



Source: Bayshore, August 2020.

Figure 10-5: Siwash North Drill Indicated Geology



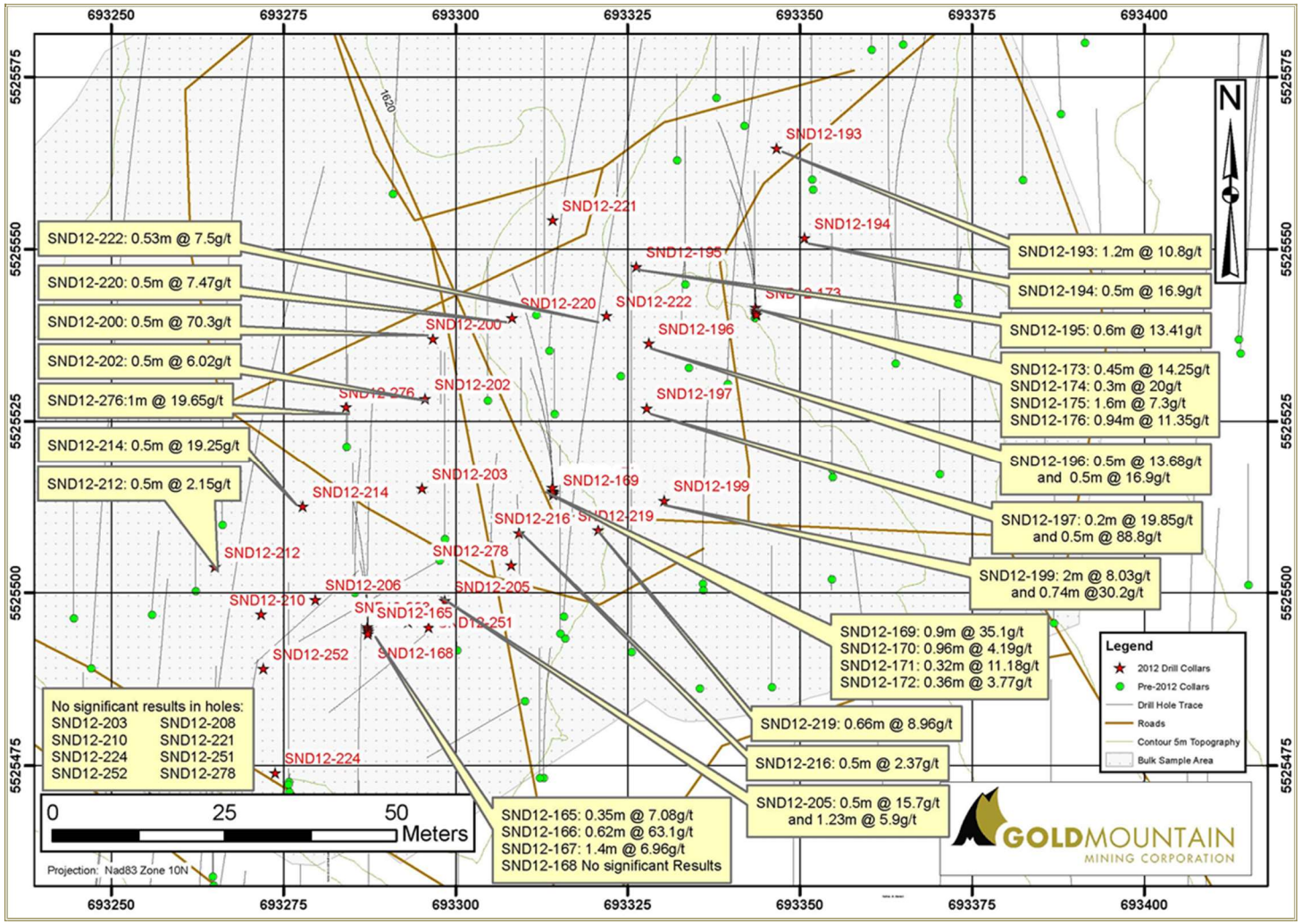
Source: Bayshore, August 2020.

Figure 10-6: Siwash North—Far West Extension Results

Bulk Sample Pit 2

Drilling in the then proposed bulk sample Pit area (Pit 2) was designed to provide fresh core to examine the tenor of veins, to provide new vein samples for metallurgical testing, and to add survey control for mining operations. The drilling was all within the quartz monzonite host rock which has been cut by the RBF trending 335°/62°. The fault lithology is a strongly clay-altered mafic(?) dyke. Quartz vein mineralization of 1000 series veins continues up to the RBF on both its hanging wall and footwall sides with little-to-no apparent offset. However, the veins east and west of the RBF are different, in that to the west the vein is moderately dipping with minor offshoots, while to the east there are four main veins with much shallower dips.

The main targets in drilling the bulk sample area were the 1000 series veins, a series of sub-parallel quartz veins including the 1100, 1200, 1250, 1300, and 1400 veins. The veins were encountered in the drill holes for the most part, as predicted with the 1300 carrying the highest grade and widest intercepts (Figure 10-7). These results have confirmed the 1300 vein as the main target in mining operations for the bulk sample. The 2012 drilling of the 1300 vein returned grades up to 88.8 g/t Au and 121 g/t Ag over a 50 cm sample interval. True vein widths range from 4 cm to 30 cm.



Source: Bayshore, August 2020.

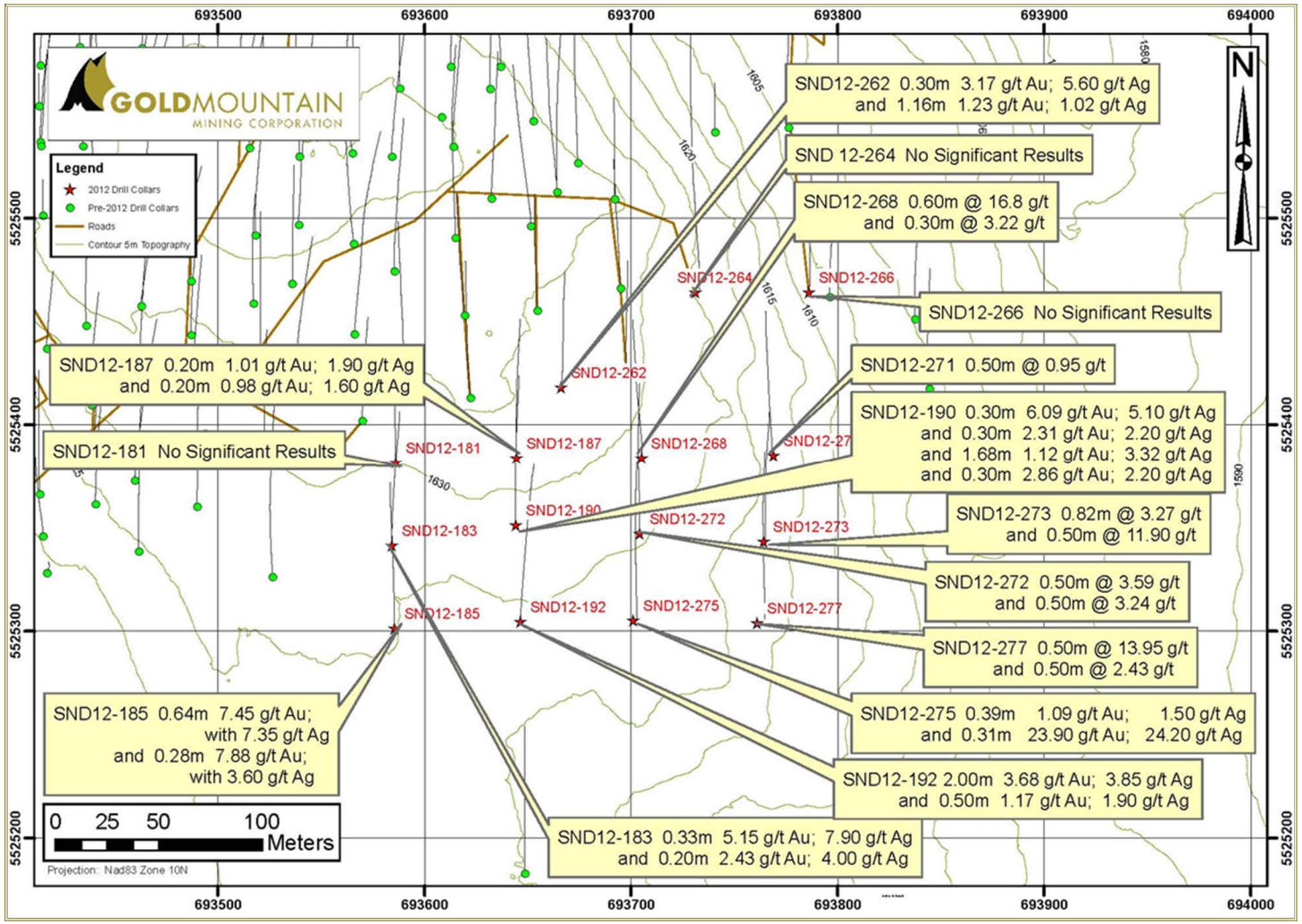
Figure 10-7: Siwash North Bulk Sample Pit Drill Results

South-East Extension

Drilling the Siwash North southeast area targeted the 1000 series veins down dip of that previously drilled at the east end of the deposit. The veins are hosted in quartz monzonite that is notably unaltered except around quartz veins; the entire area is also notably not fractured when compared to other areas within the same host rocks. The deposit continues to be open down dip and along strike, but grades are generally becoming lower (Figure 10-8). The highest grades observed in 2012 were a 31 cm sample carrying 26 g/t Au and 24.2 g/t Ag in hole SND12-275, and in SND12-277 with a 50 cm interval carrying 13.25 g/t Au and 39.3 g/t Ag. These two holes demonstrate the continuation in the down-dip direction as well as along strike, as these two holes are the furthest south (down dip) and the furthest east (along strike) in this area, and perhaps mark the start of a new higher-grade zone.

The veins in this area are notably quite consistent; most targets were intersected within 3 m of where they were predicted, and this was in an area with little previous drilling. While the veins are consistent and moderate- to high-grade, these veins were very narrow, and in vein clusters rather than as one larger vein like other areas of the Siwash North deposit. It should be noted that within these clusters, higher grades tend to correspond with a higher density of veins within each cluster.

In summary, the 84 holes drilled in 2012 were successful in confirming that the Nicola Volcanic complex can be a host rock for mineralized quartz veins on the Property, albeit as discontinuous and narrower veins that tend to carry low-to-moderate gold grades. The Pit 2 bulk sample area was confirmed to be an area of multiple pyritic quartz veins dipping 20° to 25° and grading 6 g/t Au to 88 g/t Au. It was also confirmed that the unmineralized RBF cuts, but does not displace, the mineralized veins. The drilling was also successful in confirming that the Siwash North vein continues down dip as well as along strike to the southeast.



Source: Bayshore, August 2020.

Figure 10-8: South-East Extension Drill Results

10.3 Historical and Recent Drill Program Data Reliability

A review of all historical and recent drill program data did not reveal any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. The resource estimates presented in Section 14 are based on drill holes the collars of which were re-surveyed in 2011 and 2012 by RTK GPS systems capable of sub-centimetre accuracy. Those drill collars destroyed by mining activities and unable to be re-surveyed were all close to the main control point. Comparisons between historical and recent collar surveys elsewhere in the Siwash North zone revealed only very minor discrepancies. The elevation changes between the mine grid datum and UTM datum were corrected for the holes that could not be re-surveyed.

Sampling procedures changed slightly over the years, from sampling whole core to split core to sawn core; however, a review of the protocols in place for all programs did not reveal any factors that would have impacted the results. There were minor changes in assaying protocols between the two labs used, but as discussed in Section 12 the agreement between assays from the two main labs used in blind umpire assaying was very good.

Siwash North zone core recovery within vein intersections was between 80% and 95%, and generally above 90%. The core in the intersections, though altered, was not often fractured in the box and was able to be split or sawn with good results.

10.4 Exploration Targets

A number of the deepest holes drilled at Siwash North, particularly in Zones 1300 and 2500, have intersected high-grade (greater than 5 g/t) gold mineralization. There is insufficient drilling along strike and down dip of these intercepts to establish their limits but, given the continuity of these veins elsewhere, it is considered reasonable to anticipate that this high-grade mineralization may continue into areas as yet untested by drilling. Therefore, the down-dip areas of these two veins are considered to be exploration targets.

Exploration of the South, Lake, and Gold Creek Zones by trenching and drilling has encountered mineralized veins that may be similar in style to those encountered in the Siwash North Zone. There is insufficient data to date to support a resource estimate, but the three zones are considered to be valid exploration targets.

Drilling at each of the zones has demonstrated the presence of mineralized veins over a strike length of approximately 400 m and to approximately 150 m depth down dip. The width of mineralization appears to be variable, but may range between 2 m and 5 m. Using an average bulk density of 2.7 g/cm^3, these volumes equate to a range of possible tonnages for each of the zones of between approximately 300,000 and 800,000 tonnes.

Gold assay values from the South Zone (894 assays) range from 0 g/t to 220 g/t and have an arithmetic average of approximately 0.9 g/t. Gold assay values from the Lake Zone (503 assays) range from 0 g/t to

89.7 g/t and have an arithmetic average of 1.3 g/t; those from the Gold Creek zone (251 assays) range from 0 g/t to 84.6 g/t and have an arithmetic average of 1.8 g/t.

By comparison, gold assay values from the Siwash North Zone (20,945 assays) range from 0 g/t to 1,176 g/t with an arithmetic average of 2.6 g/t. The current, pit-constrained resource estimate for the Siwash North Zone has an average gold grade of approximately 4.9 g/t at a cut-off of 0.5 g/t or approximately twice the arithmetic average. Assuming geological similarity between these zones and the Siwash North Zone, and using the simplistic relationship between arithmetic and resource estimate average grade, the South Zone may have gold grades that range between 0.9 and 1.8 g/t, the Lake Zone may have grades that range between 1.3 and 2.6 g/t, and the Gold Creek Zone may have grades that range between 1.8 g/t Au and 3.6 g/t Au. These ranges of possible tonnages and grades are summarized in Table 10-9.

Table 10-9: Exploration Target Tonnage and Grade Ranges Siwash North, South, Lake, and Gold Creek Zones

	Length (m)	Dip (m)	Min. Width (m)	Max. Width (m)	Min. Target (t '000s)	Max. Target (t '000s)	Min. Avg. Grade (g/t)	Max. Avg. Grade (g/t)	Min. Gold (oz '000s)	Max. Gold (oz '000s)
Gold Creek	500	150	2	5	405	1,215	1.8	3.6	23.4	140.6
Lake	500	150	2	5	405	1,215	1.3	2.6	16.9	101.6
South	500	150	2	5	405	1,215	0.9	1.8	11.7	70.3
Siwash North	500	125	2	5	338	1,013	5	10	54.3	325.5
Total					**1,553**	**4,658**	**2.1**	**4.3**	**106.3**	**638.0**

It should be noted that for all these zones the potential quantity and grade is conceptual in nature, that there has been insufficient exploration to define a Mineral Resource, and that it is uncertain if further exploration will result in the target being delineated as a Mineral Resource.

11 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

The following sections describe the sample preparation, analysis, and security for both historical (1989–2007) and more recent (2010–2012) drill programs. In the author's opinion the adequacy of sample preparation, security, and analytical procedures meets the standard sufficient to validate the database for the purposes of calculating a mineral resource estimate.

11.1 Historical Sample Preparation, Analysis, and Security

The first drilling on the Siwash North zone by Cordilleran was in 1989, and that year and all subsequent years to 2006 were under the supervision of Wojtek Jakubowski, P.Geo., and QP as defined in NI 43-101. The core was washed, core blocks converted to metric, and recovery plus rock-quality-designator (RQD) measurements collected. Core boxes containing mineralized quartz veins were detail photographed at five photos per box, and all other boxes were photographed at four boxes per photograph. The core was detail logged by geologists on paper, then entered into computer programs for later plotting, or logged directly into portable computers. For the years 1989 through 1996, samples of mineralized vein material consisted of the entire core, and split core was collected from all other zones sampled. From 2000 onwards, a quality assurance/quality control (QA/QC) program was included in the sampling procedure. All zones sampled were split, and every 20th sample interval the core was quartered and assigned the next sequential sample number. Blank samples were submitted to the lab at the same frequency as the duplicates. The blanks were taken from unaltered granodiorite core that contained no quartz veining. The purpose of including blanks in the sample stream was to confirm that no contamination occurred in the sampling or analysis processes. The variability of the duplicate values suggested a significant nugget effect, a finding confirmed in recent drilling. The blanks indicated that contamination was not an issue for that drill program (Jakubowski, 2002).

Mineralized quartz samples, blanks, and duplicates were shipped to and fire assayed by Acme Labs in Vancouver, BC, and for drill programs before 2000, results above 2 g/t Au were either re-assayed from a new sample split and the results averaged, or the reject was subjected to metallic screen sampling techniques and fire assayed. The drilling reports do not indicate the laboratory certified standard held by Acme at the time.

Jakubowski (2002) also described in detail the assay protocols set in place with the laboratory and were described as follows:

Samples that were expected to have significant gold content were split and half the core was submitted to the lab; in contrast to previous years when the entire core was assayed. Typically, this material consisted of quartz vein with or without wall rock, at least 10 to 15 cm thick with a minimum of 10% sulfide (or traces of visible gold). These samples were crushed in their entirety to -3/16' and coarse pulverized to -1/16". Two kg of the -1/16" material was split out and pulverized to 99% finer than -150 mesh and sieved on

a -150 mesh screen. One Assay Ton (1 AT) of the -150 mesh was assayed for gold and silver, and was combined with the weighted result of gold and silver fire assays of the entire coarse fraction, to give total gold and silver values. ICP analysis for 30 elements was also carried out on a 0.50 gm sample of -100 mesh material. This technique was referred to as the Sieve and Assay method. Selected high grade intercepts were checked by resampling from the reject and assaying for gold by the same method.

Samples which were expected to be of lower grade were comprised of half the core split along its length. This material usually consisted of quartz vein material less than 10 cm thick and less than 10% sulfide, and/or wall rock. At the lab the entire sample was crushed to -3/16", then 2kg were split out and coarse pulverized to -1/16". A 250gm split was taken and pulverized to -100 mesh. A one-assay ton (1 AT) sample was fire assayed for gold and silver. Thirty element ICP analysis was usually carried out. Higher grade intercepts were re-assayed using the Sieve and Assay method described above.

Samples that were not expected to carry high gold values, typically stringers with varying sulfide in alteration, or material of scientific interest, were split in half prior to shipping. After shipment to the lab the entire sample was crushed to -3/16", 250 gm of sample split out and pulverized to -100 mesh. A 20 gm sample of the -100 mesh material was analyzed for Au by ICP-MS using acid extraction. High grade intercepts were re-assayed using the Sieve and Assay method described above.

Samples that returned higher than expected values were assayed using the next higher confidence sampling procedure. The upgraded assays generally returned values lower than the originals. This may be due to larger sample size reducing the nugget effect. (Jakubowski, 2002)

The results of the assays were included in the assessment reports along with scans of the assay certificates. It is this author's opinion that the organization and level of detail in drill logs, assay sheets, plans, sections, and other historical documents indicates a careful and professional approach to data collection and data management.

11.2 Program Sample Preparation and Security (2010–2012)

11.2.1 Program Sample Preparation and Security (2010)

The following italicized text is a verbatim excerpt from an assessment report filed on drilling in the South Showing area (Alexander, 2011), but the QA/QC procedures and results are from the entire 2010 drill program including that on Siwash North.

Drilling and sampling on the Elk Gold Project in 2010 were completed under the supervision of Mr Robert Brian Alexander, P.Geo. and Qualified Person as defined in NI 43-101 under the executive supervision of Mr. Morgan Poloquin, P.Geo. The same procedures as described from Jakubowski (2000) and Giroux (2007) were used in 2010.

In 2010, drill core was sent for analysis at ALS Laboratory Group, Minerals Division, in Vancouver, B.C. Analysis was done using 50g fire assay with AA finish for Au (0.005-10ppm) and 50g fire assay with gravimetric finish for Au (0.05-1000ppm). ICP (Inductively Coupled Plasma) analyses for 33 elements were also performed on all samples.

One un-mineralized core (blank) and one gold standard pulps (from CDN Resource Laboratories Ltd.) were inserted into the sample stream in every thirty samples as a check of lab procedures. A duplicate sample was also taken in every thirty samples. Due to the high degree of nugget effect in the duplicate samples it was decided to send the entire second half of the drill core, instead of using a quarter split as done previously in the early portion of the 2010 program. Pieces of wood, labelled with the sample interval data, were placed in the core box in place of the missing core. Metal tags, labelled with the sample number, were stapled into the core boxes to mark the start of each sample interval. All split samples were shipped to ALS for preparation and assaying.

The analyses reported were carried out at ALS Chemex Laboratories of North Vancouver using industry standard aqua regia, ICP and fire assay techniques. Blanks, field duplicates and certified standards were inserted into the sample stream as part of Almaden's quality assurance and control program which complies with National Instrument 43-101 requirements. Intervals that returned assays below detection were assigned zero values. Composites were calculated using a cut-off grade of 0.25 g/t gold and with a maximum of 5 meters internal dilution (values below 0.25 g/t). Reported widths are intersection and not true widths unless otherwise stated. A significantly larger drill program during 2010 occurred on the Mining Lease and is not part of this assessment report however the QA/QC review includes results from that program to document issues that were experienced.

Starting in 2010, Almaden began using ALS-Chemex Laboratory (ALS) in North Vancouver as their primary laboratory. The fire assay oven had space to accommodate thirty client samples in one batch, therefore a standard, blank, and duplicate were taken one in every 30 samples. Additional QA/QC procedures were as follows:

11.2.1.1 Chain of Custody

All samples regardless of type including rock, drill cuttings, drill core, soil and stream sediment must be sealed with a plastic zip tie and stored in a secure area before shipping. Samples should only be submitted to the lab personally or through a reliable commercial courier or shipping service. Waybills of the shipments must be kept with waybill numbers sent directly to the lab and the Company at the time of shipping. Rock and drill core samples are sealed by the sampler and kept under control of a qualified person until they are shipped to a laboratory.

11.2.1.2 Sample Handling

Samplers are required to not wear any jewelry or clothing or use equipment which may contaminate the sample. All sample locations are to be geographically located at the time of sampling using the Global

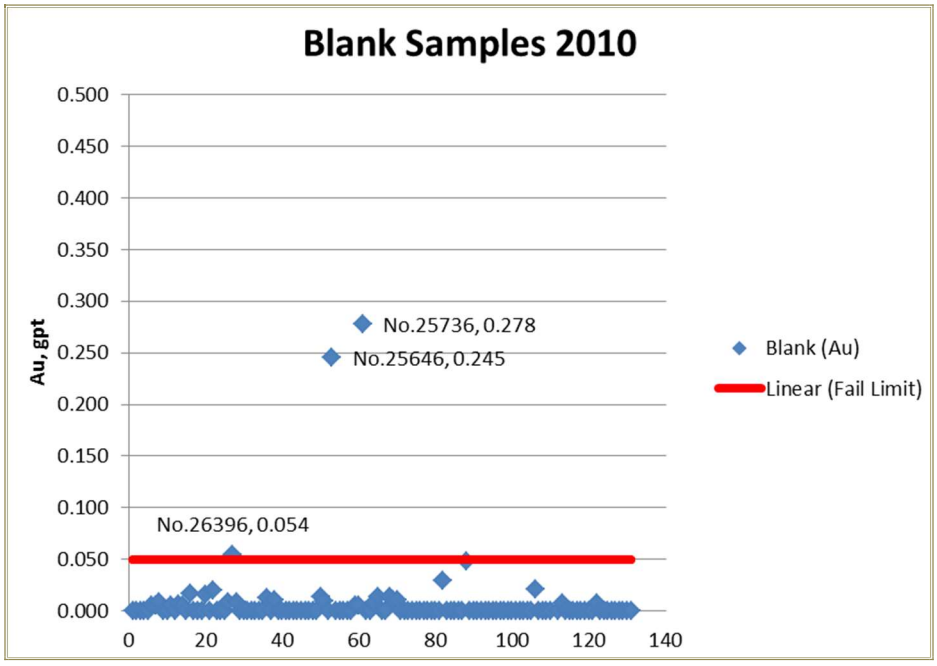
Positioning System with coordinates recorded according to the DATUM previously agreed upon to be used. Almaden has prepared standardized sample information cards for samplers to record information concerning the sample location, type and medium.

11.2.1.3 Blank Samples

Blanks were submitted into the sample stream starting in the year 2000 on a one in twenty basis and were sourced from unaltered granodiorite and quartz monzonite core pieces with no quartz veining. There are a few samples that exceed the accepted fail limit of 0.05 g/t Au but they do not appear systematic and are scattered throughout the history of the drilling campaigns. There does not appear to be any contamination problems during the sample preparation process.

In 2010 the blank material, a sample of split core known to contain very low or non-detectable concentration of gold, is inserted into the sample stream on an interval of one every 30 samples. Blanks are intended to detect possible contamination during the sample preparation phase or during the fire assay.

Figure 11-1 shows the assay results on blank samples that exceeded the tolerance limit of 0.050 g/t gold. There are a few samples that exceed the accepted fail limit of 0.05 g/t Au but they do not appear systematic and are scattered throughout the history of the drilling campaigns. There does not appear to be any contamination problems during the sample preparation process in general. An exception to this statement occurs with sample numbers 25646 and 25736, where two samples in relatively close numeric order were returned with assay values ranging between 0.245 and 0.278 g/t gold. The laboratory was contacted and reminded to be sure the equipment was adequately cleaned after each sample during the sample preparation process.

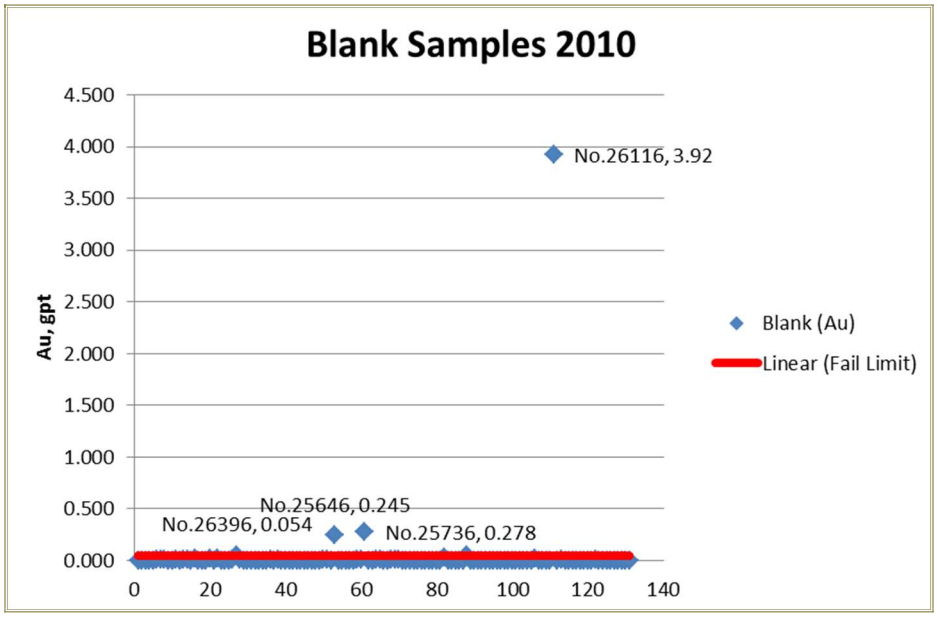


Source: Alexander, 2011

Figure 11-1: Blank Samples 2010

Source: Alexander, 2011

Figure 11-2 illustrates a blank sample (26116) that returned an anomalously high gold value of 3.92 g/t. The corresponding group of 30 samples was re-assayed. The same or similar numbers were returned. It was concluded that the problem was due to error in the selection of the field blank from the old core. The sample must have contained a small quartz veinlet that was gold bearing.



Source: Alexander, 2011

Figure 11-2: Blank Samples 2010 Including Outliers

11.2.1.4 Core Duplicates

From 2000 to 2005, core duplicates were collected every 20th sample by taking a ¼ split of the remaining ½ core and assigning it the sequential sample number to the original sample. In 2006, Almaden began sending the other half of the cut core for duplicate analysis instead of a ¼ cut of the remaining half core in the box. The frequency of insertion changed from every 20th to every 30th sample. The ¼ core to ½ core duplicates shows an apparent high bias in the results of the ¼ core samples over the ½ core samples. Results for the ½ core duplicate data compare well and do not appear to show any bias between the duplicate pair analysis results.

During the 2010 drill program the Company routinely included a field duplicate into the sample stream, spaced at 30 sample intervals. The resulting two field duplicate samples are submitted with separate sample numbers "blind" to the assay lab and separately treated as normal samples. The samples are taken randomly with no regard to rock type, geographic position or degree of alteration or mineralization. These field duplicates are then used to detect the cumulative uncertainties associated with the entire sampling and analytical process.

At the start of the 2010 season core duplicates were submitted as quarter split core. This error was corrected part way through the season, and the second half of the split core was submitted as the duplicate sample. Wooden lathe was put in the core box and labelled as a duplicate sample. A record of the core exists in the photographs routinely taken of all drill core prior to being sampled. Figure 11-3 shows a more detailed view of the duplicate samples, since most duplicates returned very low assay values. The samples

are labelled with the first value (half core) followed by the second value (quarter or half core). The first value was plotted on the horizontal axis of the graph. It is significant to take note that a greater number of samples are skewed to the right of the graph, reflecting the presence of quarter split core with lower assay values than the half core.

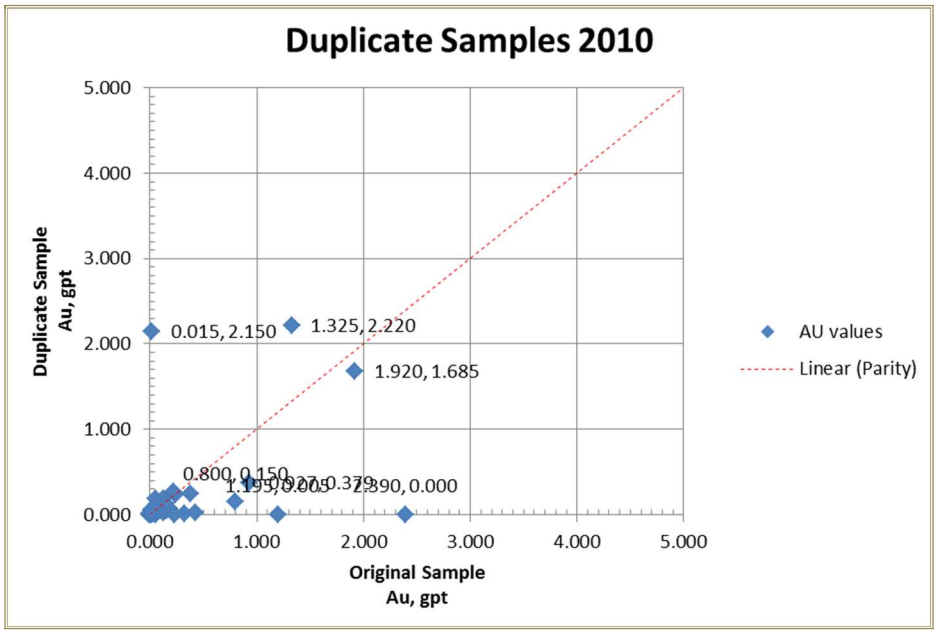


Figure 11-3: Duplicate Samples 2010 (Alexander, 2011)

Figure 11-4 shows two unusually high grade gold values that were returned as part of the random duplicate sampling. Sample numbers 18776 and 18777 returned assay values of 15.05 and 7.35 g/t, respectively from SSD10-004. Sample numbers 28545 and 28546 returned assay values of 19.30 and 1.035 g/t, respectively from SND10-004. Hole numbers SSD10-004 (South Showing) and SND10-004 (Siwash North) were drilled early in the season when the duplicate samples were composed of the first split as half core and the second split as quarter core. These samples illustrate the skewed location to the right as mentioned above.

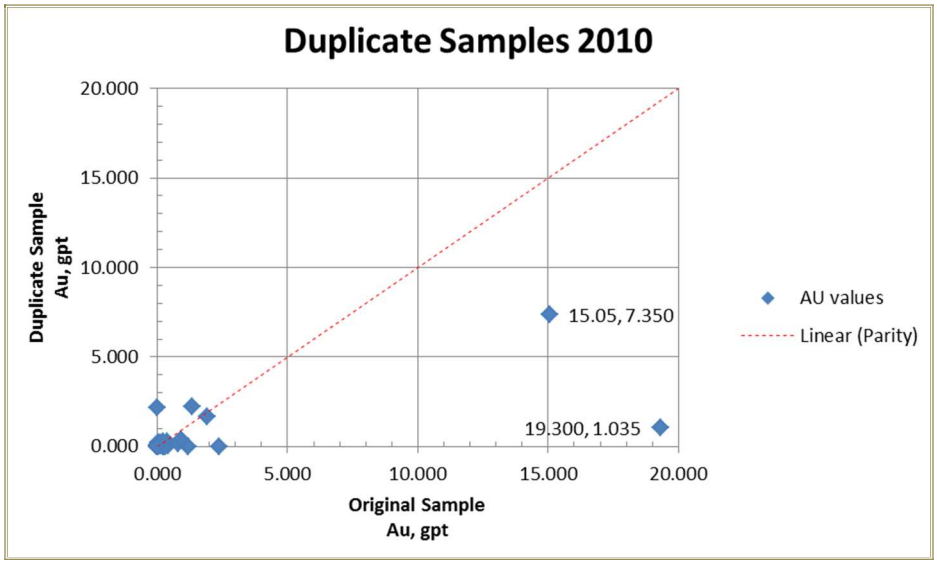


Figure 11-4: Duplicate Samples 2010 Including Outliers (Alexander, 2011)

11.2.1.5 Standards

Standards are purchased from CDN Resource Labs of Vancouver and are prepared by this professional third party lab according to industry standard and accepted methodologies. Standards are utilized to monitor the accuracy of the laboratory work. The amount of Standard Reference Material (SRM) submitted was originally 50 gram packages, which was used part way through the program. It was discovered that 100 grams should be submitted to be sure that multiple analysis of the same sample could be obtained when necessary. SRM CDN-GS-P2 was added part way through the 2010 season to fulfill a recommendation to have a low grade standard in use. A low grade standard was recommended since the cut-off grade used in calculating composites for press releases was 0.25 g/t gold. Table 11-1 lists the Standard Reference Materials used during the 2010 program.

Table 11-1: Standard Reference Material Specifications (Alexander, 2011)

Standard Reference Material Label	Au Value ± 3 SD	Upper Limit Au (g/t)	Lower Limit Au (g/t)
CDN-GS-30B	29.21 ± 1.85	31.06	27.36
CDN-ME-2	2.10 ± 0.165 g/t	2.265	1.935
CDN-GS-5F	5.27 ± 0.51	5.78	4.76
CDN-GS-2F	2.16 ± 0.36 g/t	2.52	1.8
CDN-GS-13A	13.2 ± 1.08	14.28	12.12
CDN-GS-P2	0.214 ± 0.03	0.244	0.184

Figures 11-5, 6, 7, 8, 9, and 10 show the assay results from the standards CDN-GS-30B, CDN-ME-2, CDN-GS-5F, CDN-GS-2F, CDN-GS-13A, and CDN-GS-P2, respectively. The figures are graphs showing the failure

limits for each standard based on 3 standard deviations above or below the average expected gold value, as listed above.

Figure 11-5 (CDN-GS-13A) shows the assay value for sample number 26356 falls well below the lower failure limit. This was a case where there was insufficient sample left to do an additional gravimetric assay to check the result. There was no way to fix the problem. It was decided to let the results pass, after consultation with Barry Smee. In the future, all standard reference material should be submitted in packages weighing a minimum of 150 g, so that at least three gravimetric assays can be performed. In the remainder of the 2010 program either 2 packages of 100 grams each were submitted or one package of 150 grams after standards were re-ordered.

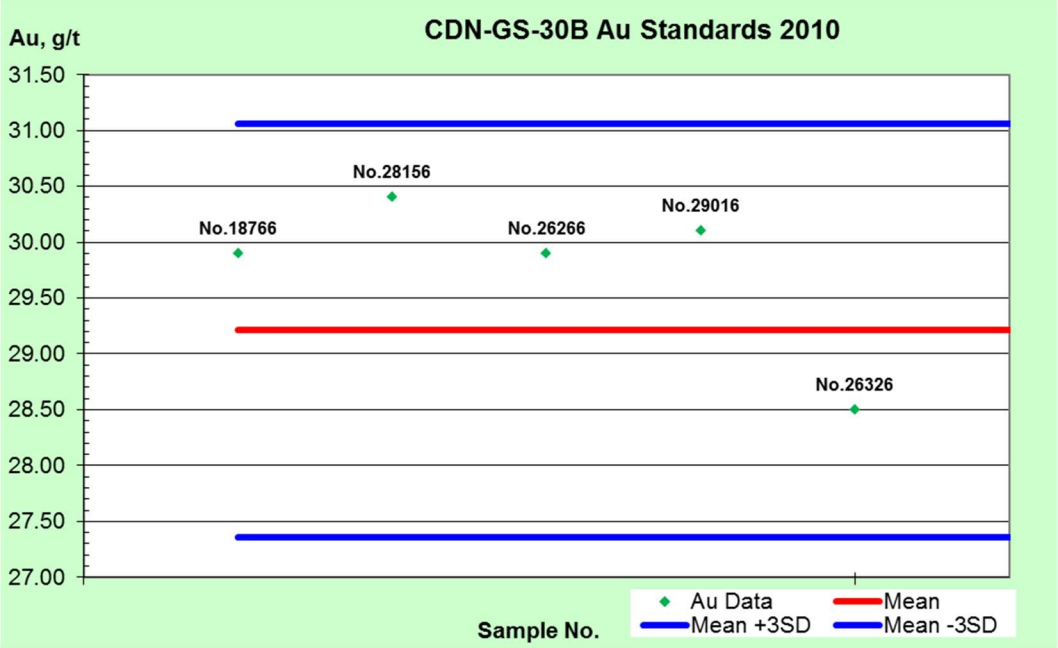


Figure 11-5: CDN-GS-30B Standard 2010 (Alexander, 2011)

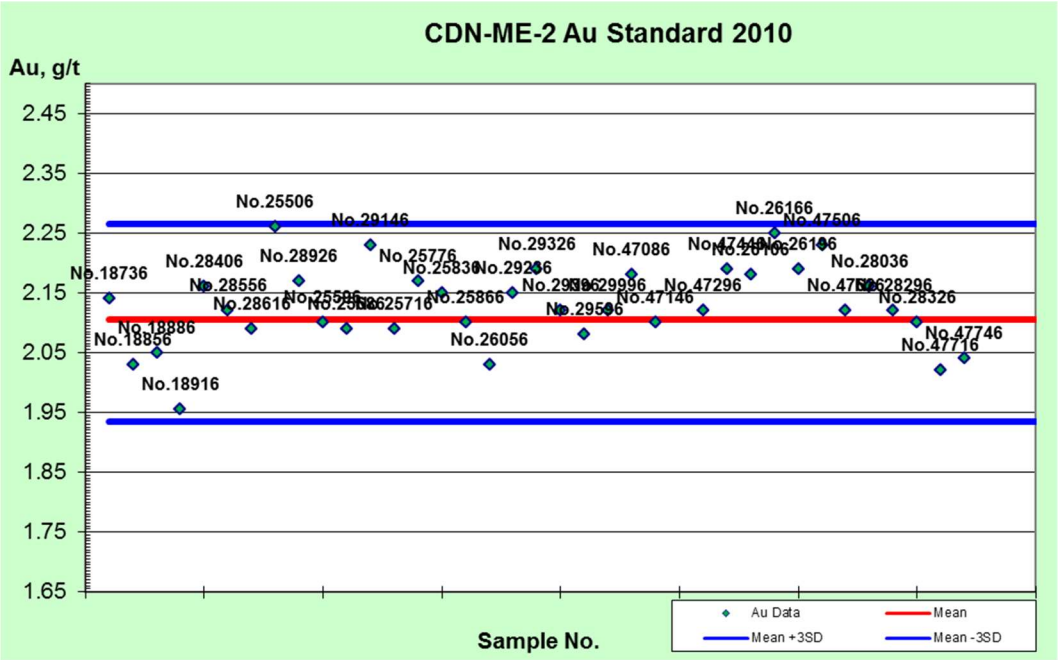


Figure 11-6: CDN-ME-2 Standard 2010 (Alexander, 2011)

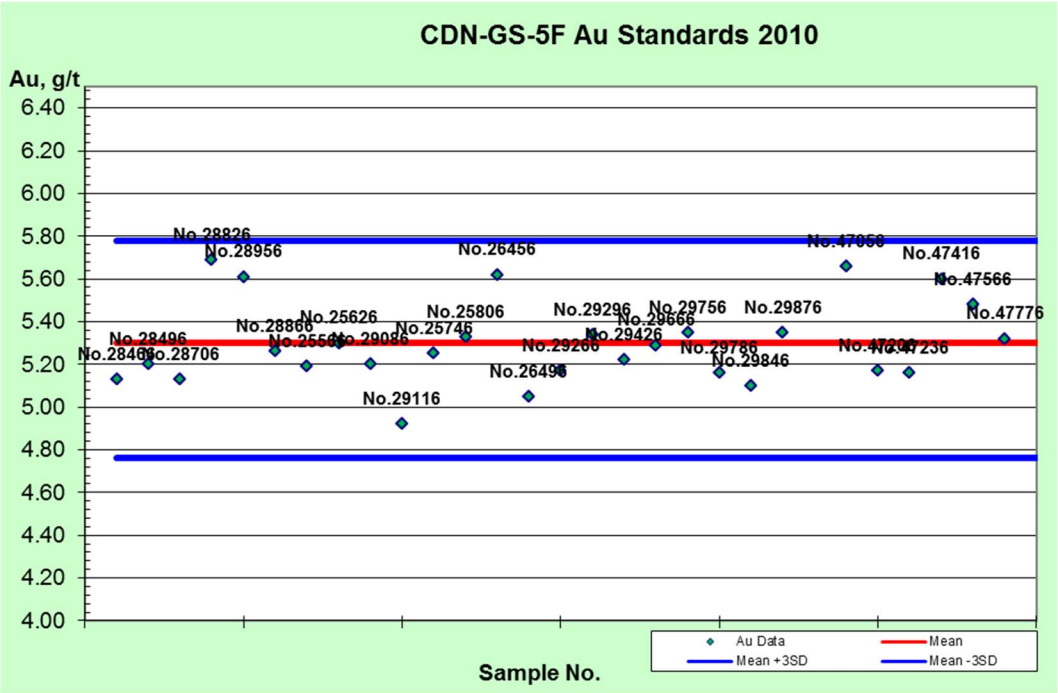


Figure 11-7: CDN-GS-5F Standard 2010 (Alexander, 2011)

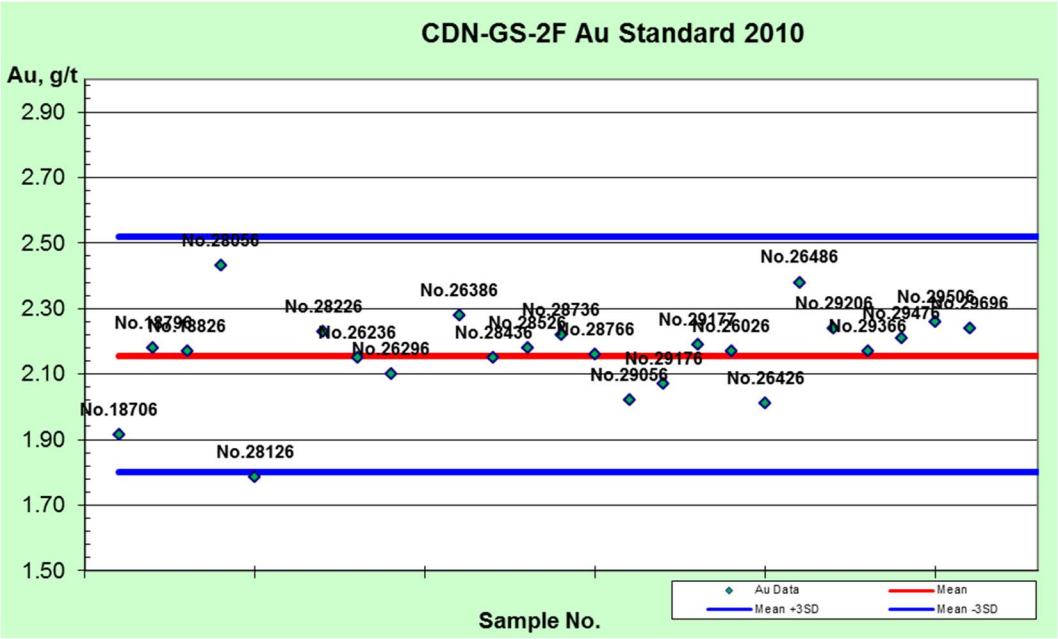


Figure 11-8: CDN-GS-2F Standard 2010 (Alexander, 2011)

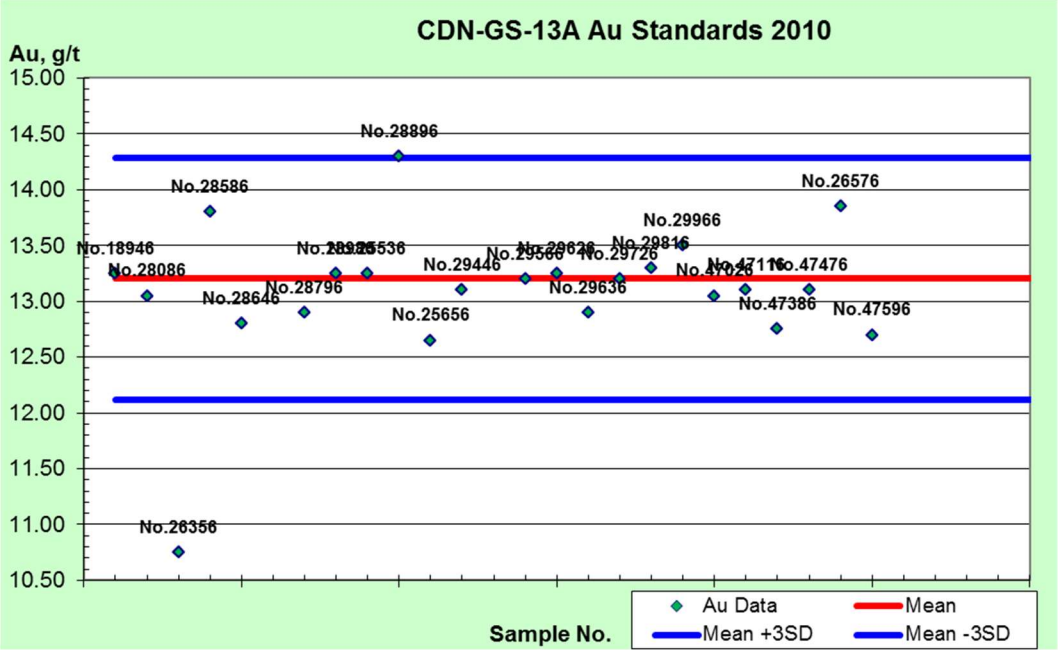


Figure 11-9: CDN-GS-13A Standard 2010 (Alexander, 2011)

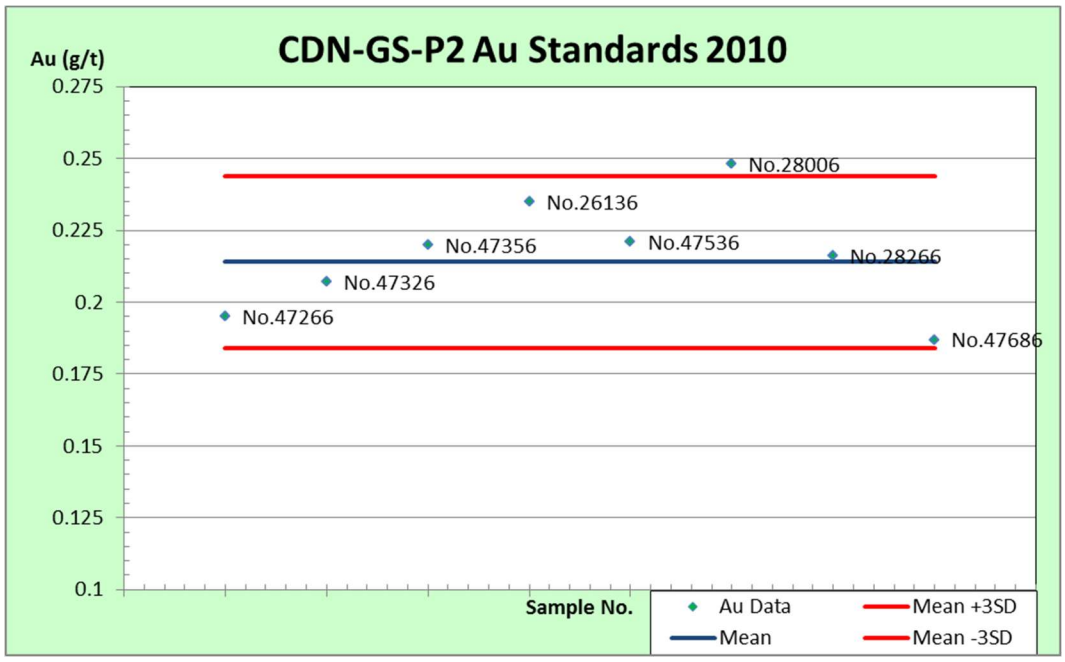


Figure 11-10: CDN-GS-P2 Standard 2010 (Alexander, 2011)

11.2.2 Program Analytical Procedures and Security (2011)

GMMC used two laboratories in 2011 to determine gold concentrations within samples. Eco-Tech Laboratory Ltd. (Eco-Tech), a division of Stewart Group, in Kamloops, BC, was the initial lab receiving core samples from GMMC. Part-way through the drill program, Stewart Group merged with ALS Chemicals and all further analyses was completed at the ALS Mineral Laboratory (ALS) in Vancouver. The following includes laboratory preparation and analysis information that has been reproduced from material provided by ALS Mineral Laboratory.

Initial drill core preparation was as described in Section 10.2.2. Following geological logging, sample intervals were laid out on the core and all information was logged into laptop computers. Next, the entire length of the drill core was photographed in groups of four core boxes to each frame. Samples were then sawn in half and one half bagged by the support staff and the remaining half left in the core box as a legacy sample available for future scrutiny.

QA/QC was completed by inserting one or more duplicate, standard, and blank samples at roughly 20-sample intervals. Duplicate samples were a quarter-core sample cut from the remaining legacy sample. Blank samples were of unaltered, unmineralized. and vein-free volcanic rock. Standards at differing levels of gold content and prepared by CDN Resource Laboratories were inserted regularly, and especially in mineralized zones. Attempts were made to match the standard gold grade with that anticipated in the vein intersection.

The plastic sample bags were closed with zip ties to prevent cross contamination, and samples were placed into numbered rice bags and stored within a locked room in the core shack. A written record was made of each rice bag's contents and a sample analysis request form along with chain-of-custody form was filled out prior to sample shipment. Samples were trucked to the preparation lab in Kamloops at regular intervals by either Greyhound bus or GMMC staff. Shipment waybills and completed chain-of-custody forms were kept on site and waybill numbers were sent directly to the lab and the GMMC head office at the time of shipping.

Lab preparation of samples was the same under both Eco-Tech and ALS. Rock samples were crushed on a Terminator jaw crusher to −10 mesh ensuring that 70% passed through a Tyler 10 mesh screen. Every 35 samples a re-split was taken using a riffle or rotary splitter to be tested to ensure the homogeneity of the crushed material. A 250 g subsample of the crushed material was pulverized on a ring mill pulverizer ensuring that 85% pass through a −200-mesh screen. The subsample was rolled, homogenized, and bagged in a pre-numbered bag. Barren gravel blanks were processed in the sample prep machines before each job and analyzed for trace contamination along with the processed samples.

The analyses preformed under both Eco-Tech and ALS used the same methods, although there was a difference in the setup and reading of the instruments used, resulting in a lower detection limit when ALS began analyzing samples. The initial analysis used an atomic absorption (AA) spectroscopy method. Procedures for analyses were as follows: prepared sample was fused with a mixture of lead oxide, sodium carbonate, borax, silica, and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead was digested in 0.5 mL dilute nitric acid in the microwave oven, 0.5 mL concentrated hydrochloric acid was then added, and the bead further digested in the microwave at a lower power setting. The digested solution was cooled, diluted to a total volume of 4 mL with demineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards. This yields an upper detection limit of 10 ppm (10 g/t) Au, so samples that exceeded this limit were then analyzed using a gravimetric method that has an upper detection limit of 1,000 ppm. This limit was never exceeded during analysis of the 2011 samples.

The gravimetric procedure was as follows: a prepared sample was fused with a mixture of lead oxide, sodium carbonate, borax, silica, and other reagents to produce a lead button. The lead button containing the precious metals was cupelled to remove the lead. The remaining gold and silver bead were parted in dilute nitric acid, annealed, and weighed as gold. Silver, if requested, was then determined by the difference in weights.

11.2.2.1 Blanks

In 2011 the blank material was a sample of split core known to contain very low or non-detectable concentrations of gold. Blanks were inserted into the sample stream on an interval of every 30 samples, and occasionally following vein samples. Blanks are intended to detect possible contamination during the sample preparation phase or during the fire assay.

As seen in Figure 11-11, blank material that was sent in was typically below detection limits and free of contamination from other samples during preparation and analyses. One sample however did have results of 0.28 g/t and was likely a product of poor cleaning of sampling apparatus either on site or at the lab, as re-analysis returned similar results. Based on the full suite of blanks submitted, this appears to have been an isolated incident of potential contamination.

It is recommended that in future programs blanks be deliberately placed immediately after a mineralized sample to detect contamination during sample preparation rather than at regular intervals. The logic is that a sample not containing gold would not show contamination in the following sample even if poor equipment cleaning were occurring. It is also suggested that the blanks not occur exactly every 30 samples to avoid recognizable patterns that may be noticed by the lab staff during preparation and analysis.

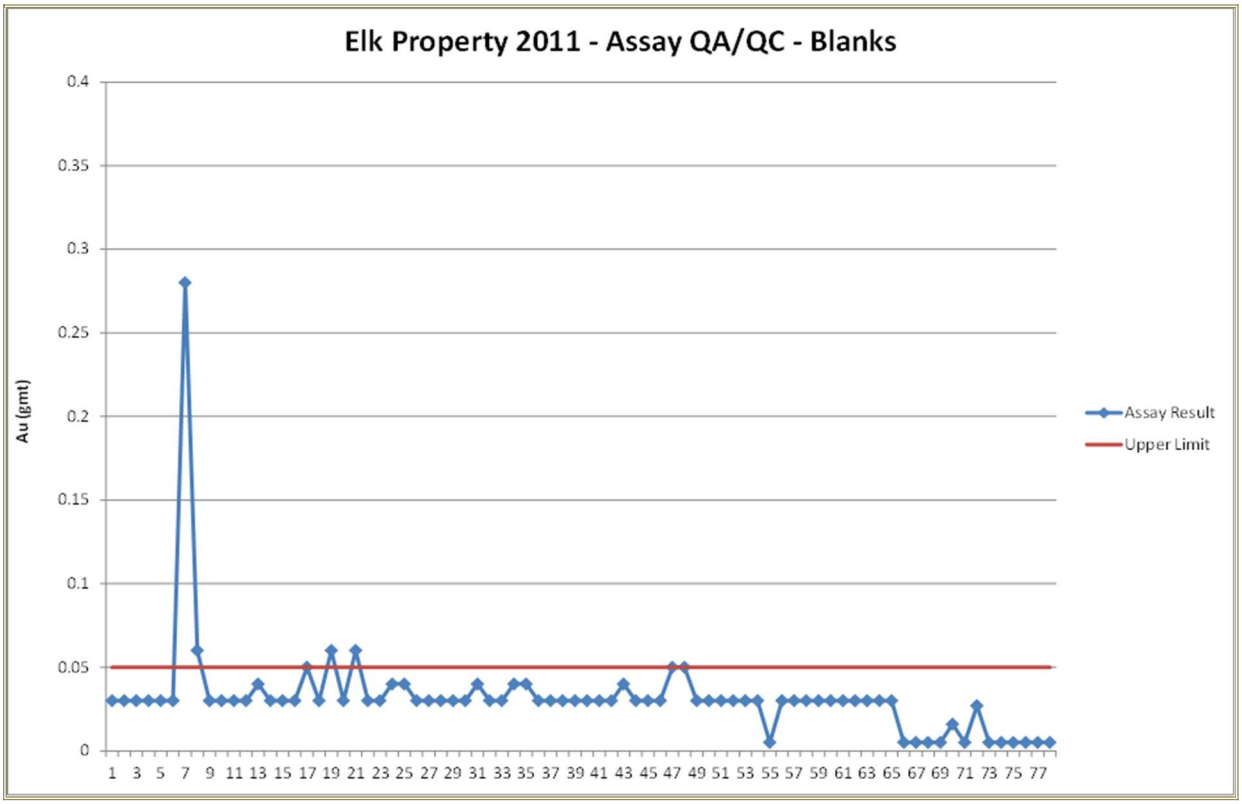


Figure 11-11: Blank Sample 2011 Results

11.2.2.2 Duplicates

During the 2011 drill program GMMC routinely included a field duplicate sample into the sample stream, spaced every 30 samples. The resulting field duplicates were submitted with separate sample numbers "blind" to the assay lab and treated as normal samples. The samples were taken randomly with no regard to rock type, geographic position, or degree of alteration or mineralization. These field duplicates were then used to detect the cumulative uncertainties associated with the entire sampling and analytical

process. To create consistent results GMMC split the core and sent each half as a separate sample. Wooden lath was put in the core box and labelled as a duplicate sample. A record of the core exists in the photographs routinely taken of all drill core prior to being sampled. Figure 11-12 is on a logarithmic scale so all samples could be included. The sample results display a general trend around the 1:1 trend-line except for a few outliers which reflects the "nuggety" nature of the gold mineralization. Generally, the higher-grade samples were further from the 1:1 trend-line, which is expected with this type of mineralization.

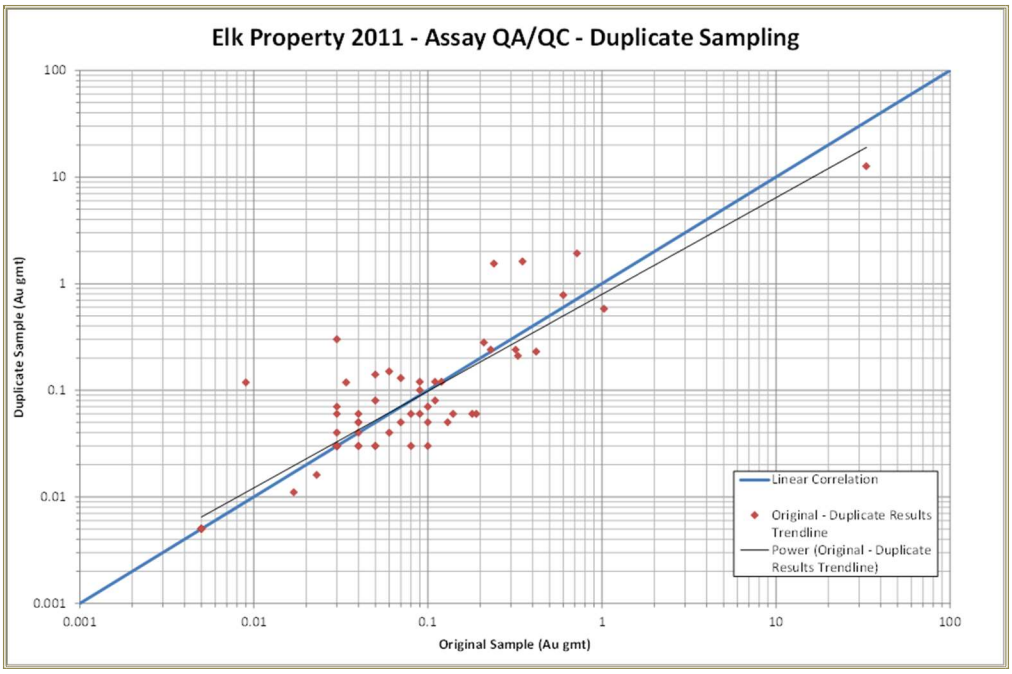


Figure 11-12: Duplicate Sample Results 2011

11.2.2.3 Standards

Standards were purchased from CDN Resource Labs of Vancouver and were prepared by this professional third-party lab according to industry standard and accepted methods. Standards are used to monitor the accuracy of the laboratory work. It was recommended in 2010 that 100 g should be submitted to be sure that multiple analysis of the same sample could be obtained when necessary. Table 11-2 lists the standard reference materials used during the 2011 program.

Table 11-2: Standard Reference Materials Used During the 2011 Program

Standard Reference Material Label	Au Value ± 3 SD	Upper Limit Au g/t	Lower Limit Au g/t	Number Used
CDN-ME-2	2.10 ± 0.165 g/t	2.265	1.935	46
CDN-GS-30B	29.21 ± 1.845 g/t	31.055	27.365	9
CDN-GS-13A	13.2 ± 1.08 g/t	14.28	12.12	32
CDN-GS-5F	5.3 ± 0.54 g/t	5.84	4.76	9
CDN-GS-P2	0.214 ± 0.03 g/t	0.244	0.184	11
CDN-GS-6A	5.69 ± 0.72 g/t	6.41	4.97	5

Figure 11-13 to Figure 11-18 show the 2011 assay results from the standards CDN-ME-2, CDN-GS-30B, CDN-GS-13A, CDN-GS-5F, CDN-GS-P2, and CDN-GS-6A, respectively. The figures are graphs showing the outliers for each standard based on 3 SDs above or below the certified gold value, as listed above. The graphs also track separately which lab analyzed each standard.

Standard CDN-ME-2 was submitted the most frequently, Figure 11-13. The results for CDN-ME-2 were all within the 3 SDs, except one sample out of the 46 sent for analysis which was 0.045 g/t Au below the lower limit of 1.935 g/t Au. This outlier could not be re-checked due to the small standard weight submitted. All other samples were within acceptable limits. It is noted that ALS Labs returned a higher average grade and Eco-Tech a lower average grade than the certified value.

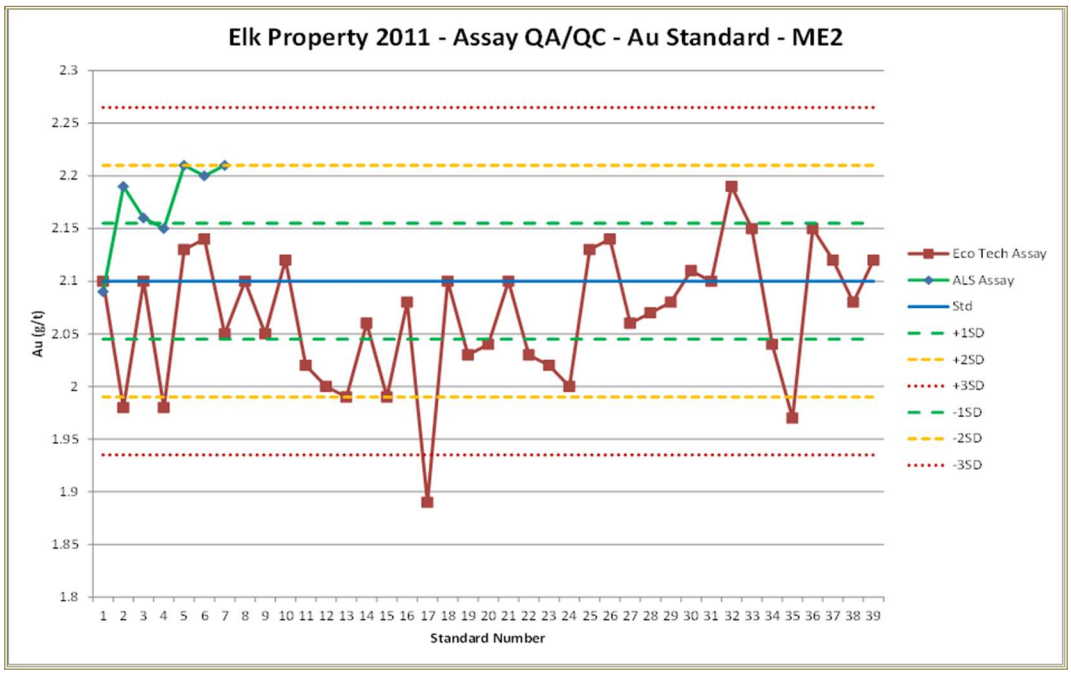


Figure 11-13: CND-GS-ME2 Standard 2011 Results

Standard CDN-GS-30B was included in the sample stream nine times (Figure 11-14). All but one sample for this standard were analyzed by Eco-Tech, and all Eco-Tech samples track above the certified value for this sample, but all were within 3 SDs. The single ALS-analyzed sample was closer to the standard's assay than any of the Eco-Tech assays.

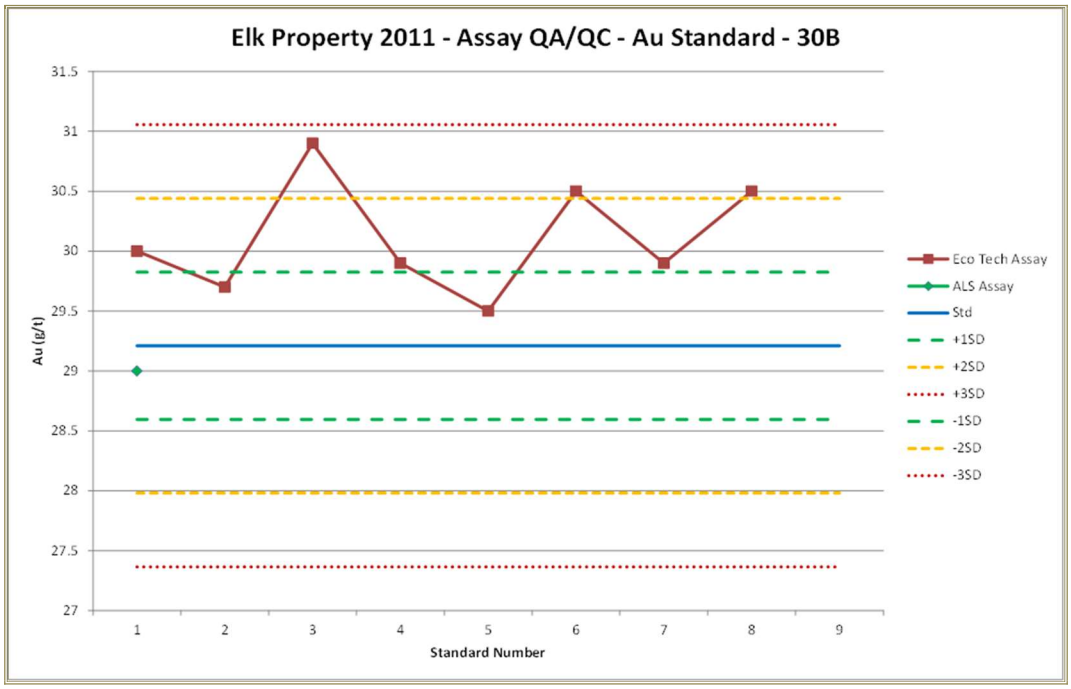


Figure 11-14: CDN-GS-30B Standard 2011 Results

The results for CDN-GS-13A had two samples out of the 32 submitted that were slightly above the 3 SDs guideline (Figure 11-15). The first was 0.02 g/t Au above and the second 0.12 g/t Au above the 14.38 g/t Au guideline. These two samples were part of significant swings in sample results for one of the early sample shipments. Assays of this standard included in shipments before and after this period display results over a much narrower range and closer to the certified standard value. It appears that any problems in the lab were solved as the season progressed. ALS Lab results were in the same wider range for this standard, but the average agreed closely with Eco-Tech analyses.

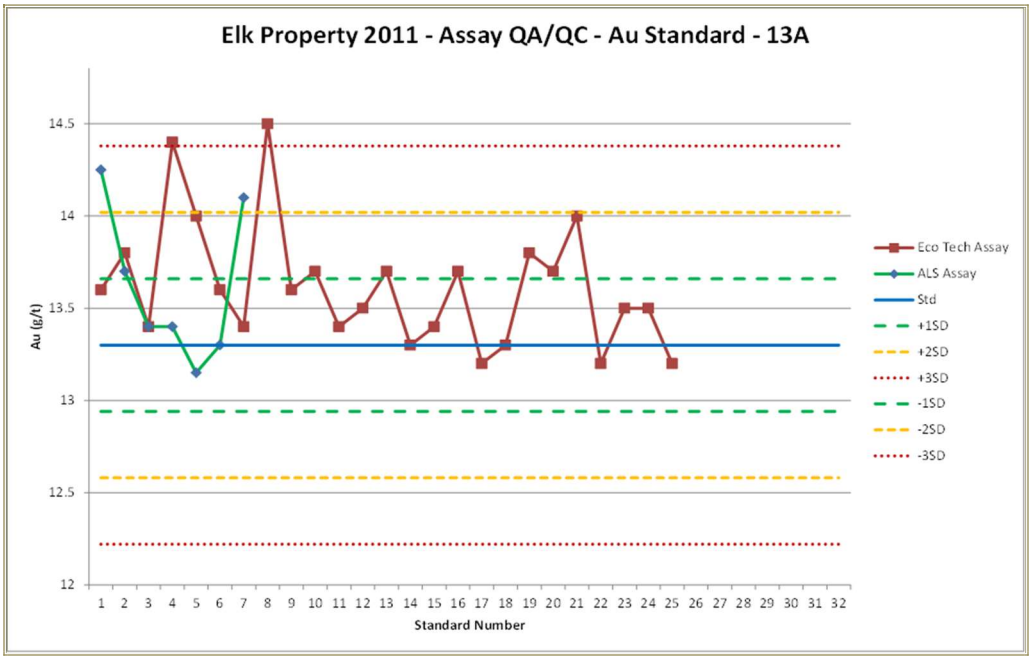


Figure 11-15: CDN-GS-13A Standard 2011 Results

The results for all nine CDN-GS-5F standards submitted were within the 3 SDs displayed above (Figure 11-16). As with Standard 13A, samples of GS-5F submitted towards the start of the season vary more widely than later on.

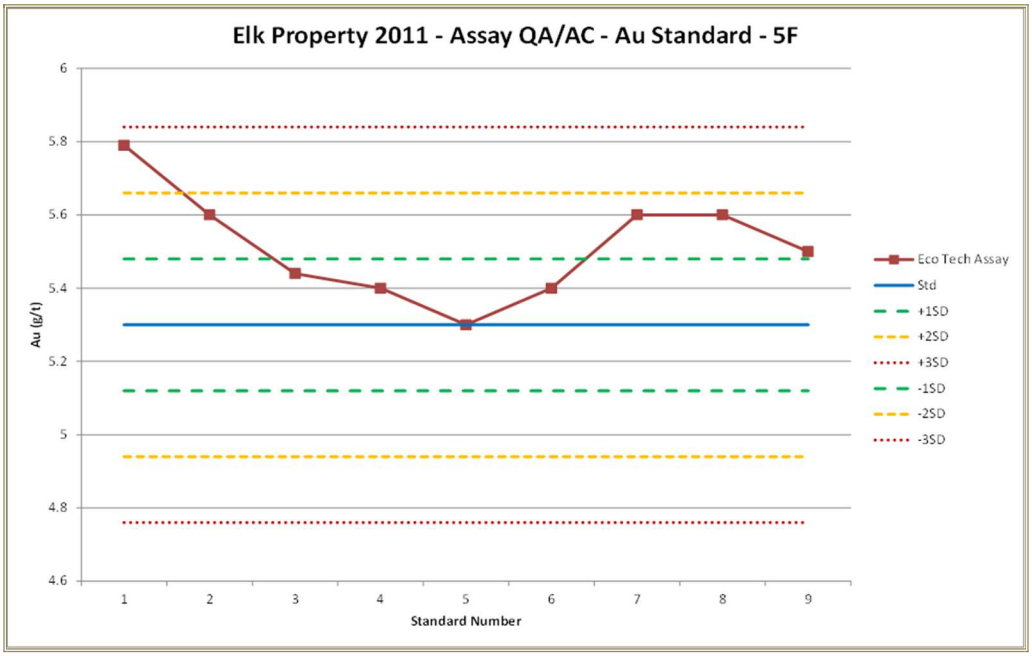


Figure 11-16: CDN-GS-5F Standard 2011 Results

Standard sample CDN-GS-P2 was submitted to both Eco-Tech and ALS labs. In total, 11 samples were submitted (Figure 11-17). The results for CDN-GS-P2 were all within the 3 SDs except for the first sample. The outlier assayed 0.026 g/t Au more than the acceptable deviation of 0.244 g/t Au and was analyzed at Eco-Tech Labs. Like standards ME2 and 30B, ALS lab results are lower and closer to certified value for the standard.

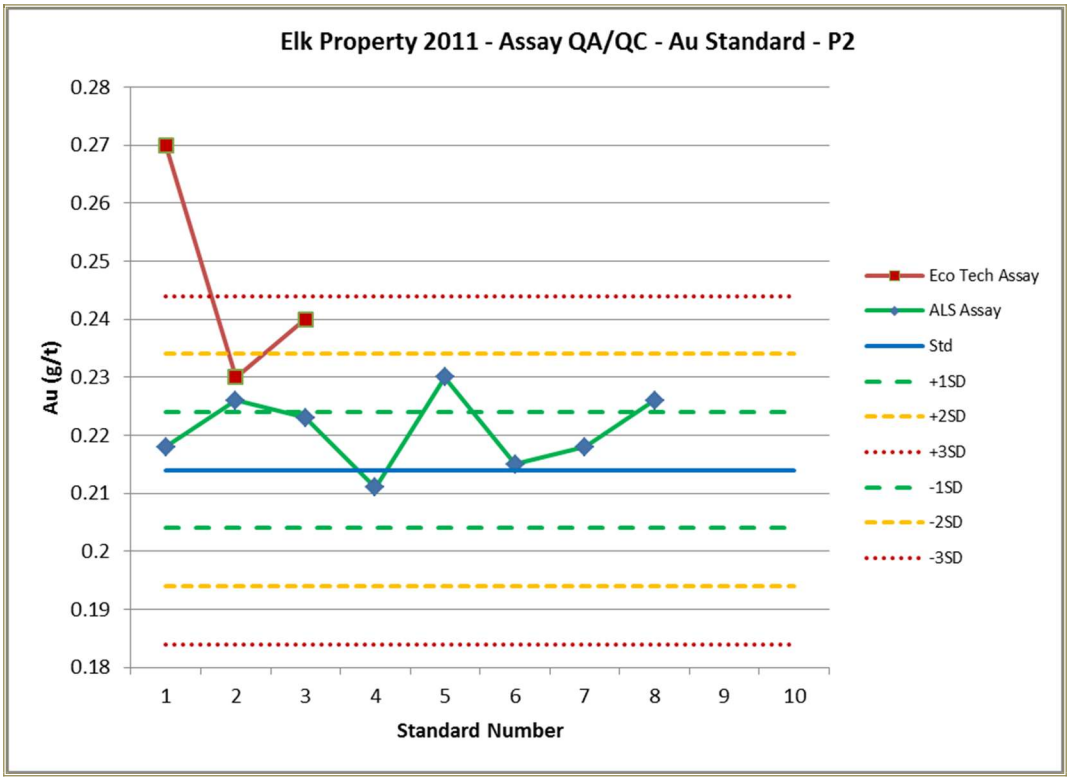


Figure 11-17: CDN-GS-P2 Standard 2011 Results

The results for all five CDN-GS-6A standards were within the 3 SDs displayed in Figure 11-18.

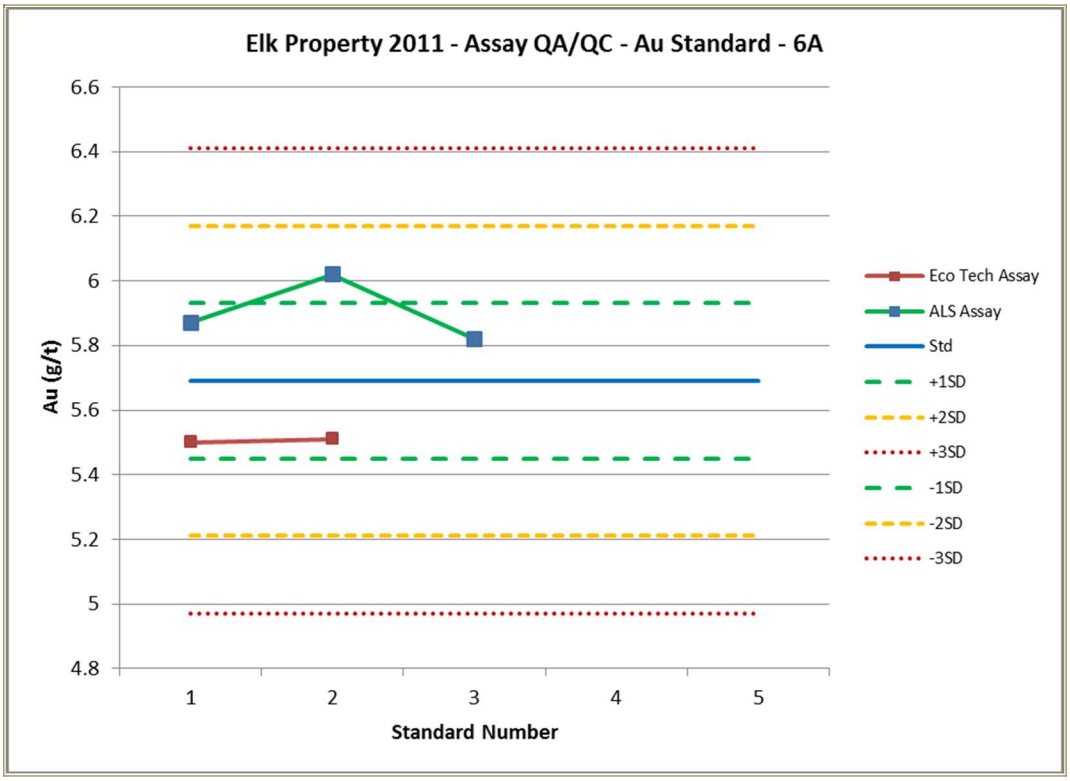


Figure 11-18: CDN-GS-6A Standard 2011 Results

In all, 5 of 112, or less than 5%, of the assayed standard samples were outside 3 SDs in the 2011 standard sampling program. Since the outliers were only very slightly outside of 3 SDs, and occurred randomly, there is no reason to believe that there were any systematic errors in the sampling or analyses procedures. Where standard samples were submitted to both Eco-Tech and ALS, Eco-Tech regularly assayed higher than ALS.

11.2.3 Program Analytical Procedures and Security (2012)

ALS was chosen for the 2012 drill program to analyze gold, silver, and other trace elements. Bagged samples were shipped directly to Kamloops, BC, to ALS's preparation facility where samples were crushed and pulverized prior to the pulps being sent to ALS's Vancouver lab for analysis. The next sections describe sample preparation, transport, and analysis. The following preparation and analysis information has been reproduced from material provided by ALS Mineral Laboratory on their website (ALS Global, 2013).

11.2.3.1 Sample Preparation

Initial drill core preparation was as described in Section 10.2.2. Sample intervals were laid out on the core and all information was logged into laptop computers. Next, the entire length of the drill core was photographed in groups of four core boxes to each frame. Samples were then sawn in half, and one half bagged by the support staff and the remaining half left in the core box as a legacy sample available for

future scrutiny. Once the sample had been taken it was placed into a new plastic bag and zip-tied closed to prevent any form of cross contamination.

11.2.3.2 Chain of Custody

At the project site, the sampler placed all samples in plastic sample bags, sealed with a plastic zip tie, and stored in a secure area which was either under constant supervision by the employee responsible for sampling, or kept locked otherwise until the samples were shipped to the laboratory. Sample bags were placed inside of numbered rice bags and logged individually before shipment. A written record of the sampler and sampling dates was kept in a sampling log, and a record of rice bag contents was kept in the sample shipment log. Copies of sample request forms and chain-of-custody documents were always kept, and originals sent with the sample shipment. Samples were transported to the lab through either a reliable commercial courier or shipping service, or trucked by company staff in site vehicles. Waybills of the shipments were kept on site and waybill numbers were sent directly to the lab and GMMC head office at the time of shipping. The lab completed the Chain of Custody form on sample receipt, a copy of which was returned to GMMC and kept on site.

11.2.3.3 Analysis

Sample preparation at the ALS prep facility is as follows:

Rock samples are crushed on a Terminator jaw crusher to −10 mesh ensuring that 70% pass through a Tyler 10 mesh screen. Every 35 samples a re-split is taken using a riffle or rotary splitter to be tested to ensure the homogeneity of the crushed material. A 250 gm sub-sample of the crushed material is pulverized on a ring mill pulverizer, ensuring that 85% passes through a −200-mesh screen. The sub-sample is rolled to homogenize it and bagged in a pre-numbered bag. Barren gravel blanks are processed in the sample prep before each job and analyzed for trace contamination along with the processed samples.

Gold fire assay was performed by ALS in North Vancouver, BC. Procedures for gold assay were as follows:

A prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica, and other reagents as required, inquarted with 6 mg of gold-free silver, fired and then cupelled to yield a precious metal bead. The bead is digested in 0.5 mL dilute nitric acid in the microwave oven, and 0.5 mL concentrated hydrochloric acid is then added, and the bead is further digested in the microwave at a lower power setting. The digested solution is cooled, diluted to a total volume of 4 mL with de-mineralized water, and analyzed by AA spectroscopy against matrix-matched standards. This yields an upper detection limit of 10 ppm Au (10 g/t), so samples that exceed this limit were then analyzed using the similar procedure with a gravimetric method finish which has an upper detection limit of 1,000 ppm, and a lower detection limit of 0.05 ppm. This 1,000 ppm limit was never exceeded by 2012 samples. The gravimetric procedure is as follows: A prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica, and other reagents to produce a lead button. The lead button containing the precious metals is cupelled to remove the lead. The remaining gold and silver bead is parted in dilute

nitric acid, annealed, and weighed as gold. Silver, if requested, is then determined by the difference in weights.

Multi-element ICP analysis was also performed on Elk Gold Project samples. Procedures are as follows:

A prepared sample is digested with aqua regia in a graphite heating block. After cooling, the resulting solution is diluted to 12.5 mL with deionized water, mixed and analyzed by inductively coupled plasma-atomic emission spectrometry. The analytical results are corrected for inter-element spectral interferences.

QA/QC procedures were performed by inserting one or more blank, duplicate, and standard samples roughly every 20 samples. Blank samples were derived from unaltered, non-mineralized, vein-free volcanic rock as selected by an on-site geologist. Duplicate samples were performed by attaching an individually numbered, empty sample bag to the sample to be duplicated. The duplicate would be taken from a second split off the crushed sample. Standards were prepared by and purchased from CDN Laboratories. These known gold standards were inserted in and around mineralized zones with the appropriate gold standard value for the anticipated assay of vein material.

11.2.3.4 Blanks

The purpose of using blanks in a diamond drill sampling program is to assure that the laboratory is following the necessary procedures to clean equipment between samples, and thus prevent contamination of samples following a high-grade intersection. To attain this goal, samples of split core reliably devoid of gold or silver were inserted into the sampling process at an interval of every 20 samples. The blank material used in the 2012 field season was a careful selection of non-mineralized Nicola Volcanic split core, known to possess very low or non-detectable amounts of gold.

The graphic analysis shown below (Figure 11-19), represents the results of the blank assays. Notably, most results are below the industry cut-off threshold of 0.05 ppm Au except for five samples analyzed near the start of the program. In total, the assays were 95% under allowable limits with the five outliers only slightly above cut off. Although one sample did run 0.13 ppm Au, after reviewing the number of assays within threshold, this assay appears to be an isolated incident of contamination and any outliers that occurred were sufficiently low enough that gold intersections would not have been unduly affected.

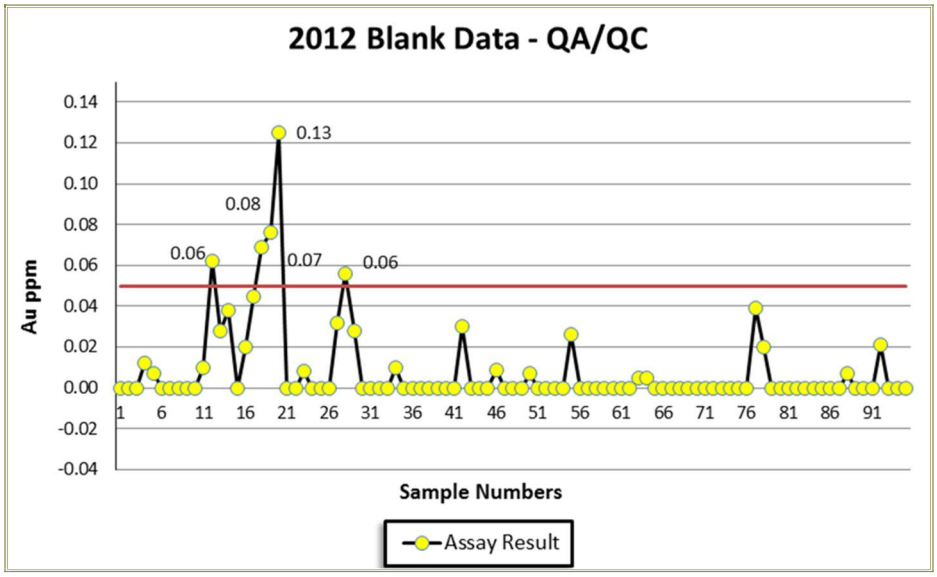


Figure 11-19: Blank Samples 2012

11.2.3.5 Duplicates

The insertion of duplicates into the sampling stream aims to scrutinize the assay repeatability (or precision) of a certain sample. In 2012 a duplicate sample was incorporated into the sampling process on an average of one every 20 samples. This is one-and-a-half times the industry standard and was used to not only test the assay lab's procedural consistency, but also as a measure of the nugget effect within mineralized Elk quartz veins.

The general sampling process involves the horizontal splitting of a predetermined length of core, where half is labelled with the sample number and returned to the box for storage. The other half is bagged with the sample number and assay ticket visible, and the bag securely tied with a zip tie. For duplicate samples a second, empty sample bag with the next sequential sample number is attached to the first sample bag, to be filled with a re-split of the coarse rejects. The samples for duplicates are chosen randomly with no regard to rock type, geographic position, or degree of alteration or mineralization. This practice ensures the analyses will be conducted randomly. For those duplicates taken within mineralized quartz veins, a greater variability of results is suggestive of a more nuggety deposit and therefore the greater the care required in resource interpretation between intersections.

Once assayed, these duplicates are graphically analyzed (Figure 11-20). Theoretically, when graphed, the results should closely follow a trend line slope of 1:1. In the figure below, the trend line coincides closely with the theoretical (1:1) trend relationship. The actual assay results have a coefficient of determination of 0.99 out of a possible 1 which suggests that the lab processes are within acceptable industry standards. Below 4 g/t Au, the duplicate results form a tight cluster surrounding the theoretical trend. Above a 4 g/t Au cut off, four of eight samples show a nugget effect suggesting that higher-grade samples contain larger individual gold particles, which are not evenly distributed throughout the host rock.

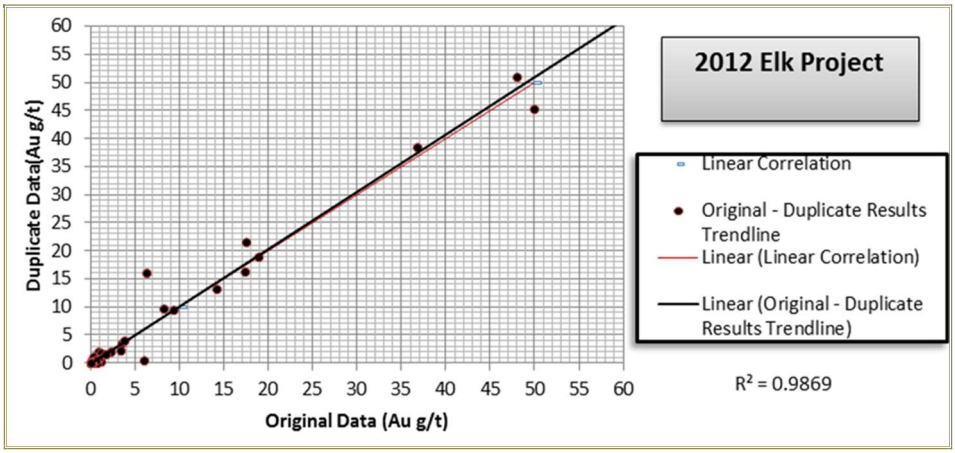


Figure 11-20: Duplicate Samples 2012

11.2.3.6 Standards

Standards are used to track drift in an assay lab's equipment and provide data by which results can be normalized for sample sets that are submitted over time. Standards were inserted at the end of mineralized runs as well as close to the mineralization zones. Standards in the 2012 field season were purchased from CDN Resource Labs of Vancouver (CDN). CDN uses multiple third-party labs to perform round-robin assays of a certain batch of mineralized samples to arrive at a certified value for the standard.

For maximum laboratory efficiency it is suggested that the sample size be upgraded from 60 g to 90 g for future programs. This increase in weight allows the lab to perform re-analysis of material if results are outside of 3 SDs from the certified value.

Table 11-3 shows a summary of the standards used during the 2012 field season. The gold values have been given by CDN, as well as the calculated SDs. Specific standards were inserted into the sample stream near mineralized veins, approximating the expected grade. Within the table, the number used is simply a count of the standards from each grade bin that were submitted.

Table 11-3: CDN Lab Standards Used in 2012

Standard	Au Value ±3 SD	Upper Limit Au g/t	Lower Limit Au g/t	Number Used
CDN-GS-13A	13.2 ± 1.08 g/t	14.28	12.12	22
CDN-GS-30B	29.21 ± 1.85 g/t	31.06	27.37	31
CDN-GS-6A	5.69 ± 0.72 g/t	6.41	4.97	31
CDN-GS-1P5D	1.47 ± 0.225 g/t	1.695	1.245	11
CDN-GS-P2	0.214 ± 0.03 g/t	0.244	0.184	B3
CDN-GS-22	22.94 ± 1.68 g/t	24.62	21.26	B6
CDN-GS-1P5E	1.52 ± 0.165 g/t	1.685	1.355	B3
CDN-GS-1P5F	1.40 ± 0.18 g/t	1.58	1.22	1

Figure 11-21 to Figure 11-24 are graphical representations of standards' assay results. Threshold lines have been added to ease depiction of SDs.

CDN-GS-13A—This standard was submitted 20 times and it can be seen within Figure 11-21 that all results for gold standard CDN-GS-13A were within the limits of 3 SDs although slightly above the theoretical actual value. There is neither a down nor up trend in the analysis throughout the program.

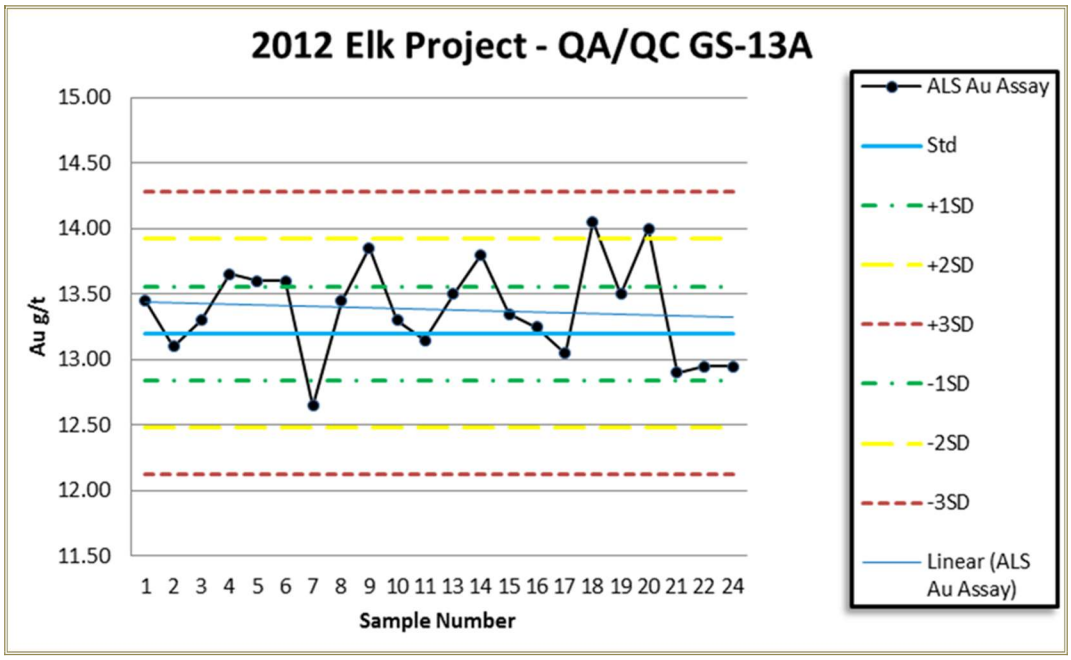


Figure 11-21: CDN-GS-13A Standard 2012 Results

CDN-GS-30B—Assay results shown in Figure 11-22 have two samples that were outside of 3 SDs. Possible reasons for the outliers are either a labelling error at CDN, a mis-recording of sample numbers in the field or some other error in the assay lab. Since standards are batch-prepared at CDN, it is unlikely that a labelling error occurred. Sample 28, at 25.5 g/t Au, and Sample 29, at 10.95 g/t Au, do not correspond with the nominal value of any other standard used at Elk, and since the received weight at the lab is consistent with standard samples these assay anomalies are therefore thought to be a lab error. These errors are isolated and part of the same sample shipment. All other sample results are within allowable SDs, but again slightly on the high side of the actual value. An even trend is recognized except as influenced by the outliers, the result of consistent lab equipment calibrations.

There was insufficient standard sample remaining at the lab to perform a re-run, hence the cause of the error remains unknown. ALS committed to finding the cause of the failures, and fresh bags of standards were submitted for inclusion with re-runs. Re-runs were consistent with original and expected assays, and the cause of the outliers remains unknown.

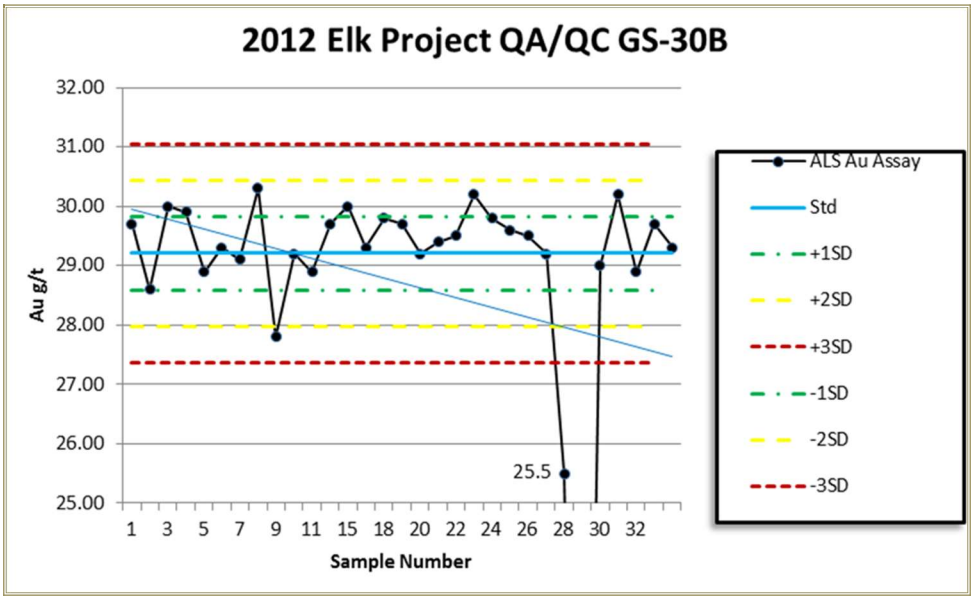


Figure 11-22: CDN-GS-30A Standard 2012 Results

CDN-GS-6A—This graph displays the results of a uniform set of assays that are all within 2 SDs, and the majority within one standard deviation (Figure 11-23), with the exception of sample 20, which had low gold values. Again, no other standard has a certified grade close to the assayed 0.107 g/t Au value, and insufficient standard material remained for a re-run. ALS investigated the error but no cause was found. It is noted that for other submitted samples of this standard the assay variation around the nominal value decreases with time.

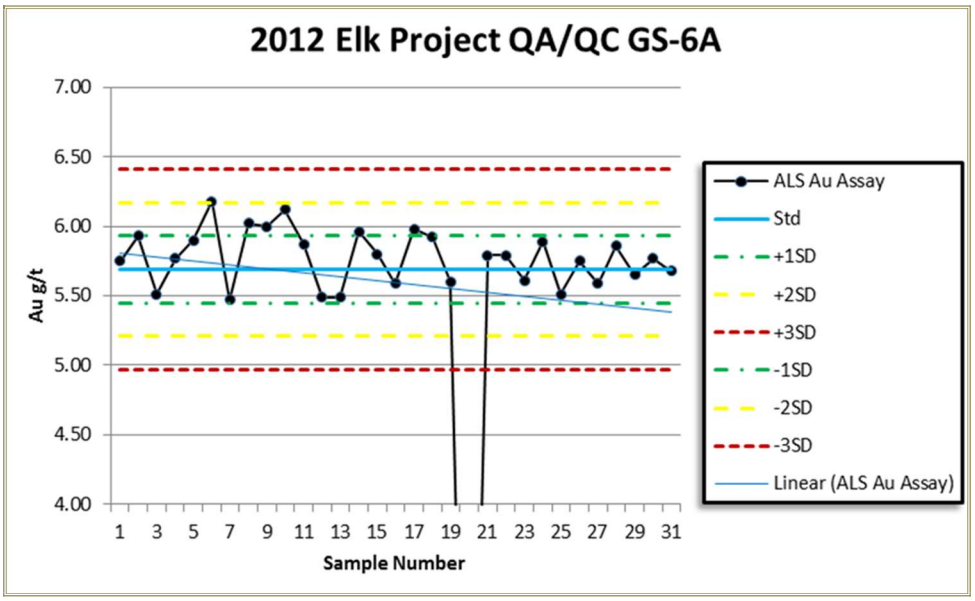


Figure 11-23: CDN-GS-6A Standard 2012 Results

CDN-GS-1P5D—All assays but one that were completed on this standard were within 1 SD (Figure 11-24), and there is no sign of instrument drift. As is the case with all other standards, assay by ALS Labs is slightly above the standard's certified value.

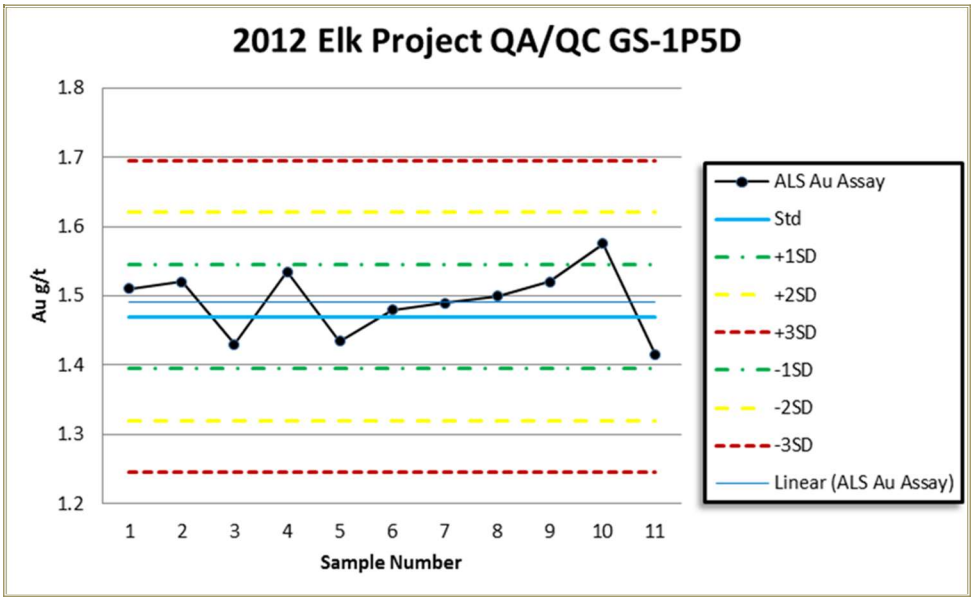


Figure 11-24: CDN-GS-1P5D Standard 2012 Results

The remaining four standards had insufficient numbers of samples submitted to define a trend, and so are omitted from this analysis. Out of 108 standard samples submitted for assay only seven were outliers beyond 3 SDs. This equates to <6.5% error in the laboratory assays. Contributing error factors may include: the mass of material being assayed, the homogeneity of the material being assayed, heavy element settling and trapping in bag folds, matrix effects due to other elements in solution, inaccurate addition of fluxes prior to firing, missed addition of silver with fluxes, as well as instrument calibration error and drift. ALS committed to further tests to identify the possible causes and thus improve their assay techniques. Due to the low failure rate, and isolated outliers, this author believes that the drill assay results are valid.

12 DATA VERIFICATION

The definition 'Data Verification' in the National Instrument 43-101 is "the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used." With historical data, this is normally performed by examining or re-entering a representative and random selection of paper-based data into a database for comparison with a digital database. Since the mid-1980s, most projects have geographical, geological, or geochemical data entered directly into a computer database, and the Elk Gold Project fits this model. For a resource estimate, this means confirming that drill data are accurate, whether it is collar locations, down-hole surveys, or sample locations and assays.

Collar locations were confirmed by independent surveyors' re-survey using RTK GPS equipment capable of sub-centimetre accuracy. The original surveying was by transit and tight metal chain from control posts, and though a few discrepancies were found, most re-surveyed drill collars in the Siwash North zone agreed with a mathematical conversion from mine grid to UTM coordinates. The datum elevation change was confirmed as correct by examining true cross sections constructed using LiDAR survey data. The drill-hole collars plotted at the topographic surface on these plots. Those collars that were destroyed by mining are thus also accepted as being within acceptable and reasonable limits.

Down-hole survey measurements in 1989 drilling were by acid tests, which do not provide hole direction and only allow dip measurement within 2° of error. Starting part-way through 1990, a single shot Sperry Sun down-hole survey instrument was used. In winter 2014/2015 a GMMC employee under the author's supervision checked the Sperry Sun photographic discs against recorded dips, and paper records of Reflex down-hole surveys were confirmed against the survey database; an excellent correlation was found.

The state of core degradation from historical Elk Gold Project programs is such that original sample locations could not be checked within the core boxes; however, a review of the raw drill and sample data gave a strong level of confidence in the drill database. For recent drilling, the sample location and assay database were thoroughly examined in spring 2014, also under the author's supervision, comparing assay results from several sources, and very few errors were found.

Analytical accuracy and precision are monitored using commercial standards, blanks, re-analysis of both coarse rejects and pulps, and re-analysis of 5% of pulps at an alternate laboratory. Six commercial standards from CDN Resource, covering a range of gold grades, were inserted into the sample stream every 20 to 30 samples. The use of six standards means that the value of the standard is effectively 'blind' to the laboratory. A blank sample was also inserted every 20 to 30 samples, such that every analytical batch will have one blank and one standard. Blank material at the Property consisted of mafic volcanic core devoid of veins.

The data records for Elk Gold Project umpire lab assaying show that 108 samples were selected in 2010 and 87 samples in 2011; however, the lab certificates for the umpire assaying could not be located in the 2010 digital records. No umpire samples were selected in 2012, as the rejects and pulps were disposed of before instructions were given directing retrieval of individual samples.

The project update report for 2010 (Alexander, 2011) discusses Almaden beginning umpire assaying in 2000. Initially every 30th sample was selected and returned to the original lab with a new sample number for duplicate analysis. The results of the resubmitted samples versus the original assays show reasonable reproducibility of the gold values, with a slight downward bias at the higher-grade range (>40 g/t Au) for the second assay. No standard reference materials (SRM) were submitted with the resubmission. It is possible that if the samples were not homogenized prior to re-assay, settling of gold grains could account for the discrepancy. Starting in 2003 Almaden began sending umpire samples to a separate laboratory, and Alexander (2011) reports "reasonable reproducibility of the gold values between the laboratories." These data could not be located in the records for independent verification; however, the graphs included within Pooley et al. (2011) have been reproduced below (Figure 12-1 and Figure 12-2). Both lab tests produced similar looking graphs, possibly because of gold grains settling in the pulp packets.

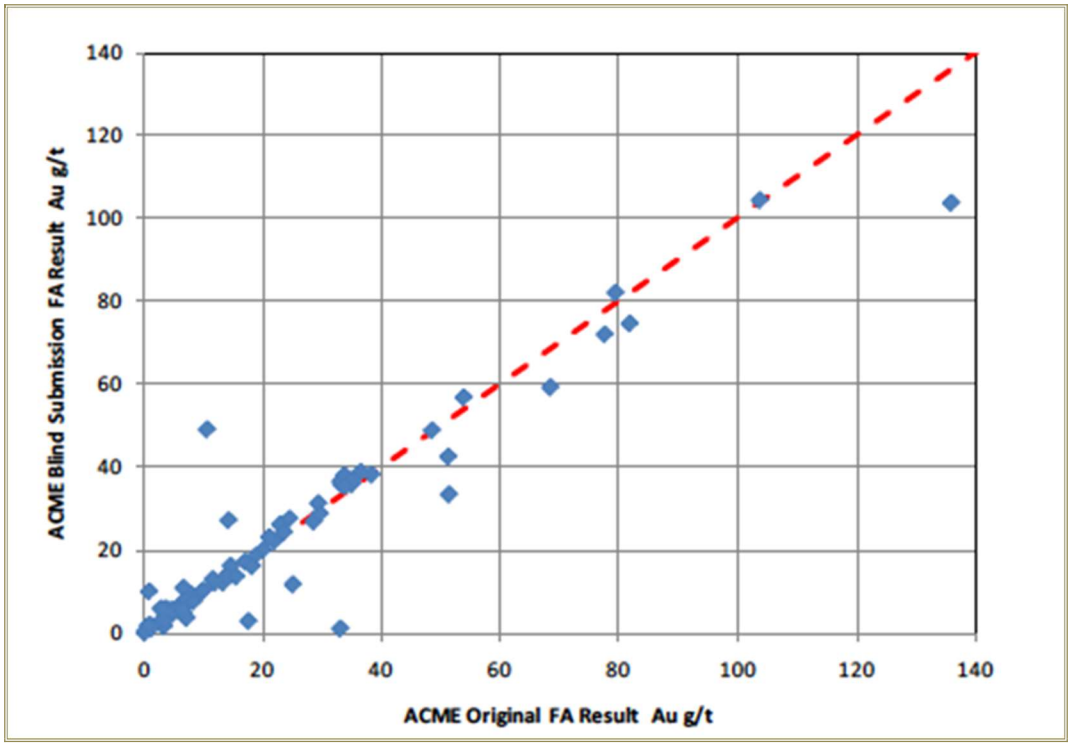


Figure 12-1: Blind Re-Assay at Acme (Pooley et al., 2011)

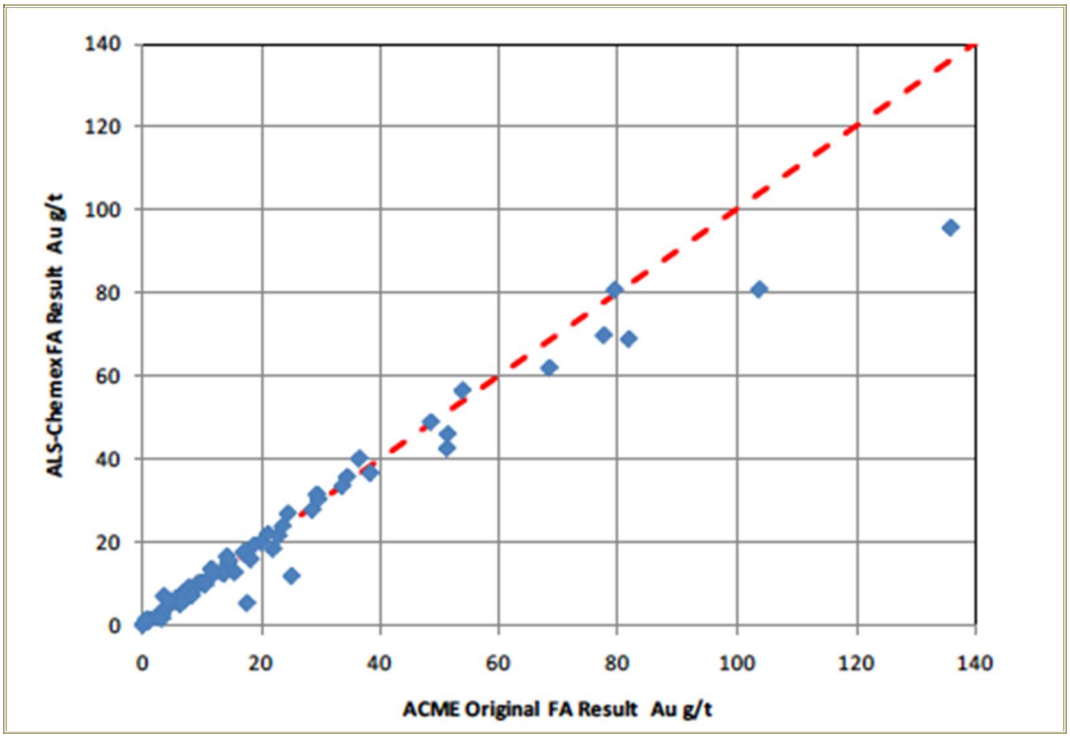


Figure 12-2: Umpire Lab Checks by Almaden as Reported in Pooley et al. (2011)

Umpire lab checks on 2011 drilling were completed by re-assaying 88 samples selected from all sample shipments and results ranges. ALS was directed to pull specific pulps from 2011 sample batches that were transported by courier to Acme Labs in Vancouver for gold fire assaying using a similar assay package as ALS. A chart of the comparative results is shown in Figure 12-3. Insufficient sample remained for three of the samples submitted to Acme, so the chart displays the results of the 85 samples that were re-assayed.

The agreement between ALS and Acme was good considering the nuggety nature of the deposit. A best-fit linear line shows the agreement to be 0.81. In general, for samples grading above 40 g/t Au the ALS assay was higher than that assayed by Acme with four exceptions, and in one of those cases the Acme assay was 2.5 times greater (456.5 vs. 182.0 g/t Au).

The laboratory check samples show reasonable reproducibility of the gold values between the two laboratories. The data show a slight bias in the results, with the Acme assay values being slightly lower than the ALS results at gold grade ranges above 40 g/t Au. This is the reverse of umpire assaying conducted by Almaden between 2003 and 2007 where Acme was the primary lab and ALS (Chemex) was the umpire, suggesting that other processes such as gold settling within the pulp packets may be causing the discrepancies at higher grades.

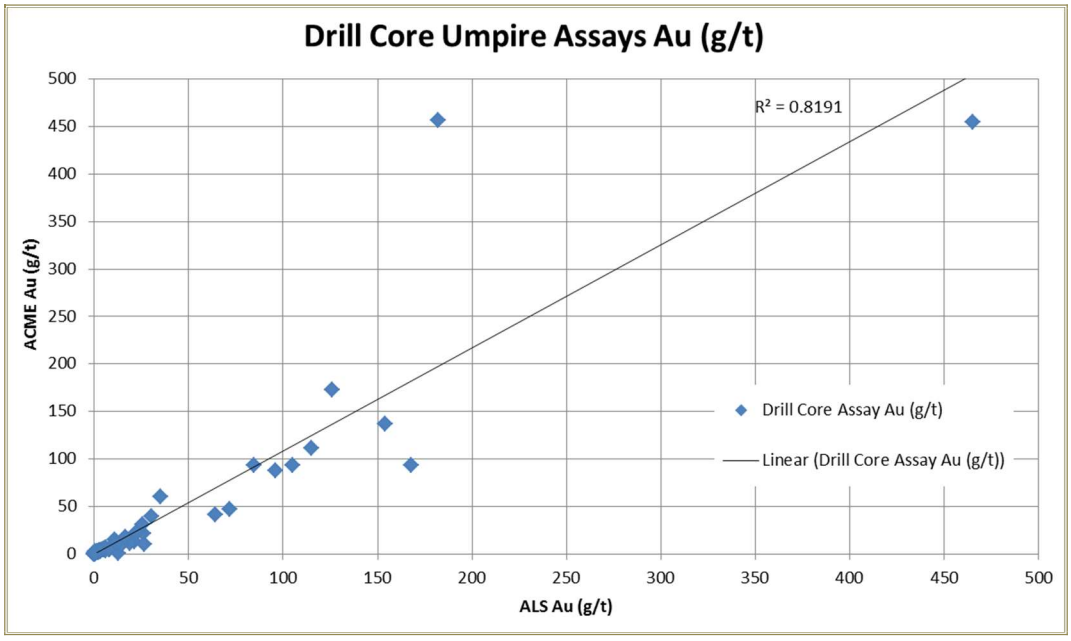


Figure 12-3: Scatter Plot of ALS Original Fire Assay vs. Acme Laboratory Check Fire Assay (Au g/t)

A review of all data inputs to the drilling database, both historical and recent, has allowed a sufficient level of confidence to include the drill database in the Mineral Resource estimate. The vein systems at the Elk Gold Project are well understood and are predictable on a section-level scale. The geological 3-D model was rechecked at great expense of time and effort, and any errors in snapping were corrected. While refinements are still possible, the solids model is the best interpretation available at present. Changes in proposed mining techniques would necessitate a review and possible reinterpretation of the solids model. Given the number of drill holes and veins on the Property, changes to the geological model would require a significant investment of labour, though recent advancements in software packages are improving the modelling procedure.

It is recommended that future drill programs use the QA/QC procedures adopted in the 2012 drill program, and request that assay standards be made larger by the standards supplier such that more than one assay can be made from the material provided. It is recommended that duplicates be made from a re-split of the coarse reject, as this ensures that the duplicate assay material is drawn from the same volume of rock.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Historically, several metallurgical test programs between 1990 and 2014 were completed on drill core, grab samples, and bulk samples from the Elk Gold Project. In addition to the test programs, bulk samples with head grades in the range of 10 g/t Au were shipped to the Kinross Kettle River Mill near Republic, Washington, in five lots and processed in a Whole-Ore Leach (WOL) circuit recovering approximately 95% of the gold.

- Placer Dome Inc., Elk Claims Project—*Gravity and Cyanidation of Bulk Samples, Report No. 1* (Issued: February 1990)
- Bacon Donaldson, Preliminary Metallurgical Testing of Siwash North Gold Deposit. File Number: MN2-041 (Issued: April 1992)
- Brenda Process Technology, *Gravity and Flotation Testwork* (Issued: August, 1992)
- Nickel Plate Mine at Hedley, BC (Issued: 1992)
- Andre LaPlante (McGill University), *The Processing of Drill Core Rejects from Fairfields* (Issued: April, 1993)
- G&T Metallurgical Services Ltd. (G&T), Preliminary Metallurgical Evaluation of Samples from the Elk Project, KM2121 (Issued: April 2008)
- G&T Metallurgical Services Ltd., *Letter Report—Metallurgical Testwork Program—Elk Project*, KM2325 (Issued: November 2008)
- G&T Metallurgical Services Ltd., *Metallurgical Evaluation of Composites from the Elk Project*, KM2522, (Issued: May 2010)
- G&T Metallurgical Services Ltd., Preliminary Metallurgical Evaluation of Samples from the Elk Project, KM4263 (Issued: July 2014).

The testwork and tests completed are summarized in Table 13-1.

Table 13-1: Testwork Summary

Date	Laboratory	Test Program	Sample Location	Composite Name	Head Grade			Testwork			
					Au (g/t)	Ag (g/t)	S (%)	Hardness	Gravity	Flotation	Cyanide Leach
Feb 1990	Placer Dome		Surface Trenching	SN89-07	46.4	77	-	-	X	-	X
			Surface Trenching	SN89-12	46.3	152	-	-	X	-	X
			Surface Trenching	SN89-13	107.0	49	-	-	X	-	X
			Surface Trenching	SN89-14	15.3	13	-	-	X	-	X
Apr 1992	Bacon Donaldson	MN2-041	Drill Core – 3 holes 97, 107, 108 SND9197-17-20, SND91107-16-18, SND91108-12-16	T1 Heads	130	99	-	10.9 kWh/t	X	X	X
Aug 1992	Brenda Mines		48-115		210	-	1.4	-	X	X	-
			48-135		48.8	-	0.3	-	X	X	-
			51-110		112	-	-	-	X	X	-
			Test Pit Comp.		125	-	-	-	X	X	X
			48 Comp.		134	-	-	-	-	-	-
Apr 1993	McGill—Laplante				119	-	-	-	X	-	
Apr 2008	G&T	KM2121	505, 506, 508-512	Master Comp.	11.5	42		-	X	X	X
			505-1,2,3	Comp 1	6.05	14		-	X	-	X
			509-10,19, 510-3	Comp 2	7.5	32		-	X	-	X
			511-3,5,6	Comp 3	4.58	6		-	X	-	X
			509-14,15,16	Comp 4	9.62	29		-	X	-	X
			509-15,17,19	Comp 5	14.6	28		-	X	-	X
			508-9, 509-13,18	Comp 6	13.3	36		-	X	-	X
			506-5,6	Comp 7	21.9	27		-	X	-	X
			506-6,8,11	Comp 8	47.1	69		-	X	-	X
Nov 2008	G&T	KM2325			11	12		-	X	-	-
May 2010	G&T	KM2522	Comp. 1-4, 506-7,12,13, 507-6	Low Grade	5.7	13.3	2.3	13.4 kWh/t	X	X	X
			Comp. 5-8, 505-3, 506-5, 507-7, 508-8	High Grade	14.4	25.1	3.45	14.1 kWh/t	X	X	X
Jul 2015	ALS	KM4263	1300 Vein	Comp 1 - HG	89.7	83	7.9	-	X	X	X

Notes: ALS = ALS Mineral Laboratory
Source: K-Met Consultants Inc. (K-Met) (2020)

13.1 Summary of Early Testwork (1990—1993)

The historical testwork summarized in this section was completed before 2008.

13.1.1 Placer Dome (1990)

Four trench samples were collected from site with head grades ranging from 15 to 107 g/t Au. The test program included gravity concentration and cyanide leach. The samples were concentrated using spiral concentrator and Wilfley table with recoveries to the final table concentrate ranging from 3.69% to 20.59%. The table tailings were leach with cyanide for 24 hours resulting in gold extraction ranging from 86.2% to 96.0% at low cyanide consumptions.

13.1.2 Bacon Donaldson (1992)

Bacon Donaldson (BD) studied a single 15 kg sample of crushed drill core rejects with head grades of 130 g/t Au and 99 g/t Ag. Two flowsheets were investigated: F1- gravity concentration followed by flotation of the gravity tailings and C1 -gravity concentration followed cyanide leach of the gravity tailings.

The following was also reported:

- Bond ball mill work index (BWi) of 10.9 kWh/t
- An inductively couple plasma ICP analysis of the feed
- Acid-base accounting (ABA) analysis of the feed.

The gravity concentration followed by rougher flotation of the gravity tailings recovered 99.5% Au overall at a combined grade of 1,050 g/t Au and 98.8% of the silver. The Au recovered to the gravity concentrate was 62.7% with an additional 36.8% recovered to the rougher concentrate at a grind size of 80% passing (P_{80}) 130 µm and a rougher mass pull of 11.9%.

An analysis of the concentrate indicated that it was mainly pyrite with less than 1% other sulphides. The free gold particles ranged from 1 µm to 50 µm.

The second flowsheet achieved a total gold extraction of 98.3% Au with 63.2% of the gold recovered to the gravity concentrate. Overall silver extraction was 91.6% with 30% Ag recovered to the gravity concentrate. The test parameters included a total leach time of 48 hours at 500 ppm NaCN and cyanide consumption was 1.07 kg/t.

The ABA analysis reported 2.97% S, with an acid potential of 92.8 kg/t, significantly exceeding the neutralization potential of 18.8 kg/t. From a permitting perspective, this metallurgical sample is PAG. However, after recovering the pyritic flotation concentrate, the flotation tailing was sufficiently depleted of pyrite so that it was non-PAG.

13.1.3 Brenda Process Technology (1992)

Brenda received five samples, including at least one panel sample from open pit mining that was producing high-grade mineralized material for direct shipping to the Noranda Horne smelter. The samples were all high-grade samples ranging from 58 g/t to 210 g/t Au.

Brenda reported gravity concentration followed by flotation recovered up to 91.7% of the gold with grades ranging from 1,000 g/t to 3,000 g/t Au. The gravity recovery ranged from 24.7% to 45.3% and overall recovery was between 73.7% to 91.7%.

Five tests were performed on the “test pit composite” to investigate grind sensitivity at grinds sizes ranging from 74 µm to 150 µm. The gold recovery was between 88.0% to 91.4% with no correlation noted as the grind size decreased.

Eight bottle roll cyanide leach tests were performed, all on the high-grade test pit composite with head grades from 76.1 g/t Au to 121.8 g/t Au. The first three tests used the "standard" bottle roll procedure, but very poor liquid/solid separation characteristics, because of "clay," resulted in an extremely high retention of pregnant solution in the leached residue. Subsequent testing used what was described as the carbon-in-leach (CIL) procedure; however, they were more similar to a Carbon-in-Pulp (CIP) test, since carbon was added after the 24-hour sampling interval. The "carbon" procedure was used to facilitate removal of the leached gold into a conveniently assayable product. The last five tests were run without gravity concentration at grind sizes ranging from P_{80} ranging from 74 to 150 µm for 24 hours at a 700-ppm cyanide concentration. Gold extraction ranged from 98.2% to 98.6% with leaching essentially completed in 24 hours. The average cyanide consumption was 3 kg/t.

13.1.4 Andre LaPlante (1993)

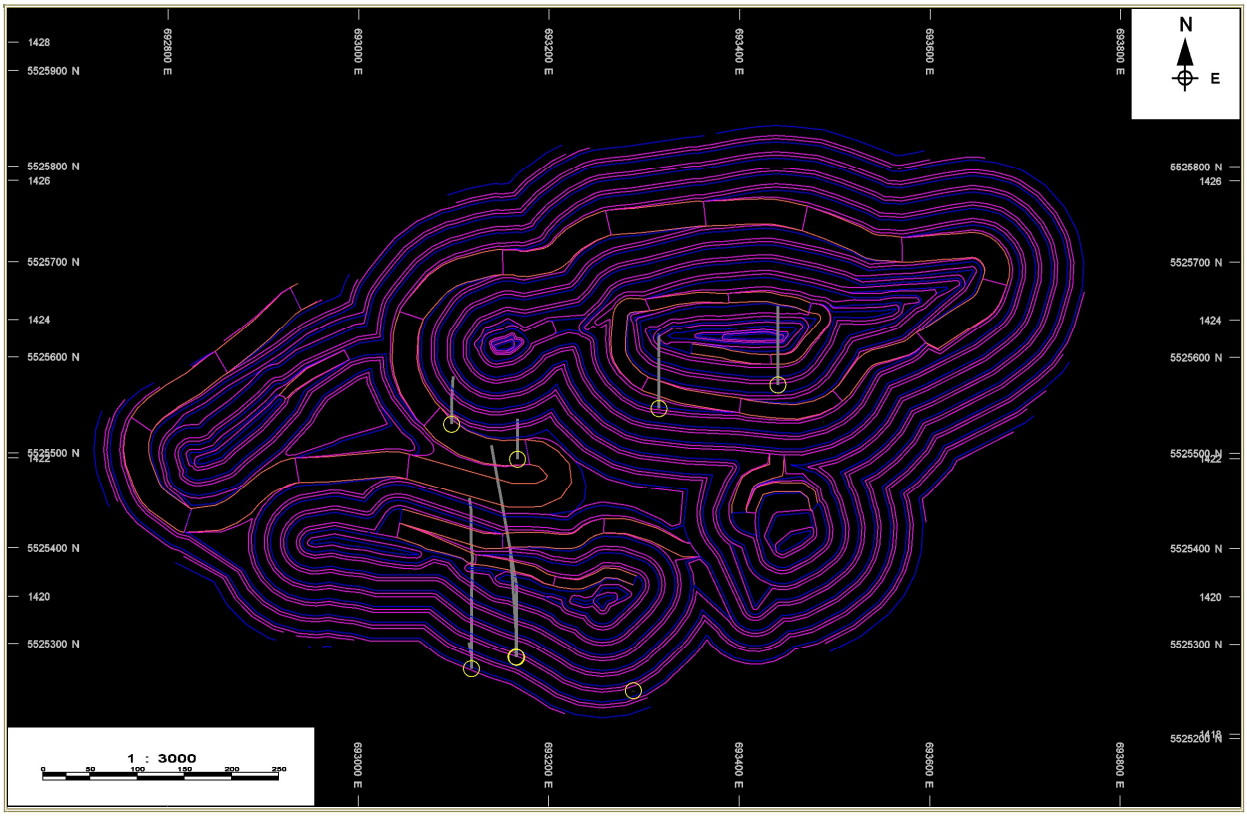
The test program included a series of gravity concentration tests to compare the Knelson vs Falcon concentrators. In both cased approximately 60% of the gold was recovered to a gravity concentrate at a coarse grind size of 39% passing 74 µm.

13.2 Summary of Most Recent Testwork (2008—2014)

The following sections summarize test Programs KM2121, KM2522, and KM4263. The most relevant results are from Programs KM2121 and KM2522 which were completed on composites in the range of the average annual head grades for gold based on the mine plan, and are the focus of the following sections. Test Program KM4263 completed by ALS investigated gravity concentration, rougher flotation, and cyanide leach on high grade material.

13.3 Sample Selection KM2121 and KM2522

Nine metallurgical holes, SND07-500 series, were drilled in 2007, and 76 samples from the drilling program were sent to G&T in 2008. The samples were used to create a master composite and eight variability composites for the 2008 KM2121 test programs, and high- and low-grade composites for the 2010 KM2522 test programs. The drill-hole locations are shown in Figure 13-1. The locations of samples collected for previous test programs could not be identified.



Source: AKF Mining Services, August 2020

Figure 13-1: Metallurgical Drill-Hole Locations 2007

Drill core samples were blended to create the composites listed below in Table 13-2 and Table 13-3.

Table 13-2: Summary of KM2121 Sample Selection

Master Composite 1

Sample	Wt (kg)
505-2	0.5
505-3	1.5
506-5	0.5
506-6	0.5
506-7	3.5
506-9	0.4
508-13	0.5
508-14	0.5
508-5	1
509-10	0.5
509-15	2.7
509-18	3.4
509-3	1.3
509-6	0.9
510-3	0.5
510-4	0.5
511-2	1.1
511-5	0.5
511-6	1.2
512-10	3.4
512-11	0.5

Composite 1

Sample	Wt (kg)
505-2	0.5
505-3	2.0
505-4	7.5

Composite 2

Sample	Wt (kg)
509-10	4.0
509-17	4.0
510-3	2.0

Composite 3

Sample	Wt (kg)
511-3	4.3
511-5	3.3
511-6	2.4

Composite 4

Sample	Wt (kg)
509-14	2.2
509-15	4.0
509-16	3.8

Composite 5

Sample	Wt (kg)
509-15	7
509-17	2
509-19	2

Composite 6

Sample	Wt (kg)
509-13	1.8
508-9	4.2
509-18	4.0

Composite 7

Sample	Wt (kg)
506-5	3.0
506-5	5.0
506-6	2.0

Composite 8

Sample	Wt (kg)
506-11	2.0
506-6	4.0
506-8	4.0

Source: G&T KM2121 (2008)

Table 13-3: Summary of KM2522 Sample Selection

Sample	Wt (kg)		Sample	Wt (kg)
Low Grade Composite			***High Grade Composite***	
KM2121 Comp. 1	6.1		KM2121 Comp. 5	7.0
KM2121 Comp. 2	6.2		KM2121 Comp. 6	6.1
KM2121 Comp. 3	6.5		KM2121 Comp. 7	6.0
KM2121 Comp. 4	5.7		KM2121 Comp. 8	5.8
506-7	4.4		505-3	1.0
506-12	0.7		506-5	0.4
506-13	4.0		507-7	9.7
507-6	3.6		508-8	3.9
Total	**37.2**		**Total**	**39.9**

Source: G&T KM2522 (2010)

13.4 Head Assays

The head assays for the composites from Programs KM2121 and KM2522 ranged from 4.58 g/t Au to 47.10 g/t Au (Table 13-4).

Table 13-4: Composite Head Assays

Composite Name	Assays					
	Au (g/t)	Ag (g/t)	Fe (%)	S (%)	Cu (%)	C (%)
KM2121						
Master Composite 1	11.5	45.0	4.1	4.5	-	-
Composite 1	6.1	14.0	-	-	-	-
Composite 2	7.5	32.0	-	-	-	-
Composite 3	4.6	6.0	-	-	-	-
Composite 4	9.6	29.0	-	-	-	-
Composite 5	14.6	48.0	-	-	-	-
Composite 6	13.3	36.0	-	-	-	-
Composite 7	21.9	27.0	-	-	-	-
Composite 8	47.1	69.0	-	-	-	-
KM2522						
Low Grade	5.7	13.1	3.6	2.3	0.03	0.40
High Grade	14.4	25.1	4.5	3.5	0.14	0.34

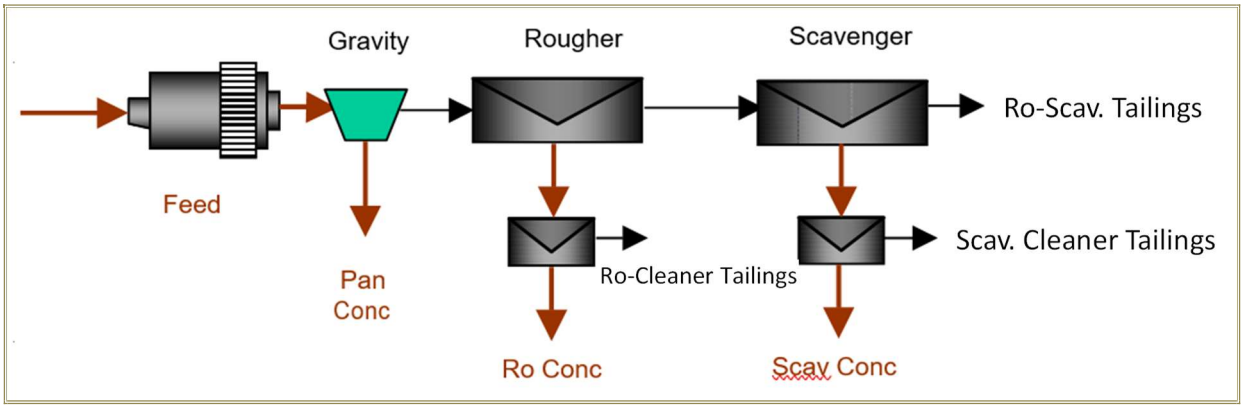
Source: G&T KM2121 (2008) and KM2522 (2010)

13.5 G&T KM2121 (2008)

Program KM2121 investigated flotation and 48-hour whole-ore leach tests. The whole-ore leach tests were carried out at varying grind sizes ranging from 80% passing (P_{80}) 40 µm, to P_{80} 300 µm, and cyanide concentrations of 250 ppm to 1,000 ppm, with and without gravity concentration prior to the leach circuit. Flotation was completed with gravity before the rougher flotation.

The master composite was used to complete two gravity flotation tests; the result at a P_{80} of 102 µm recovered on average 44% of the gold in the gravity circuit, and 48% to the rougher concentrate, for a total combined recovery of 92% with a grade of 151 g/t.

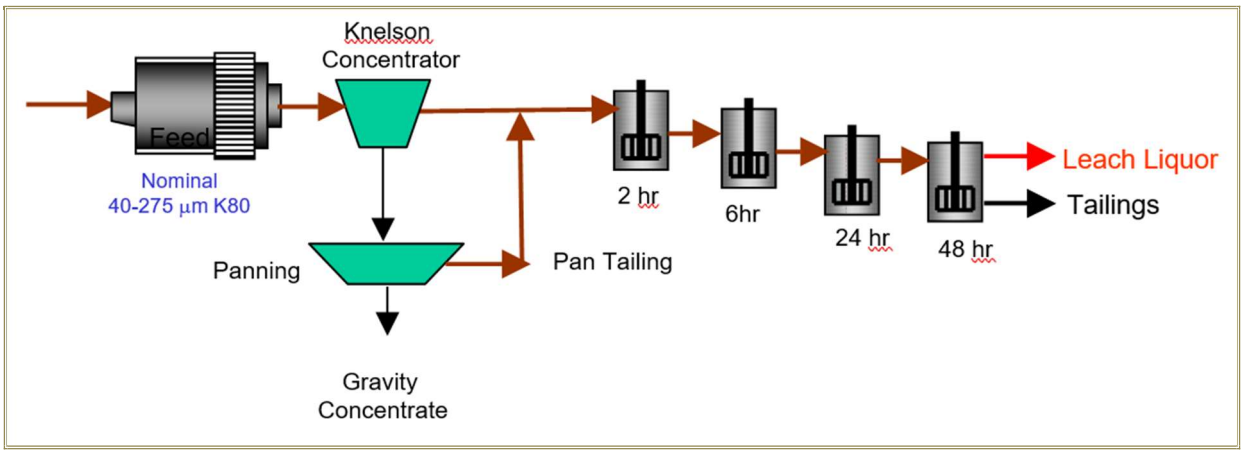
The Master composite was tested using the cleaner flowsheet shown in Figure 13-2. Combined gravity and cleaner concentrates recovered 95% of the gold at a grade of 126 g/t, with a mass recovery of 9%. The tests determined that bulk sulphide flotation achieved high recovery using a standard reagent regime of methyl isobutyl carbinol (MIBC) frother, and potassium amyl xanthate (PAX), at natural pH. Silver recoveries were not reported for the flotation testwork.



Source: G&T KM2121 (2008)

Figure 13-2: Cleaner Flotation Flowsheet

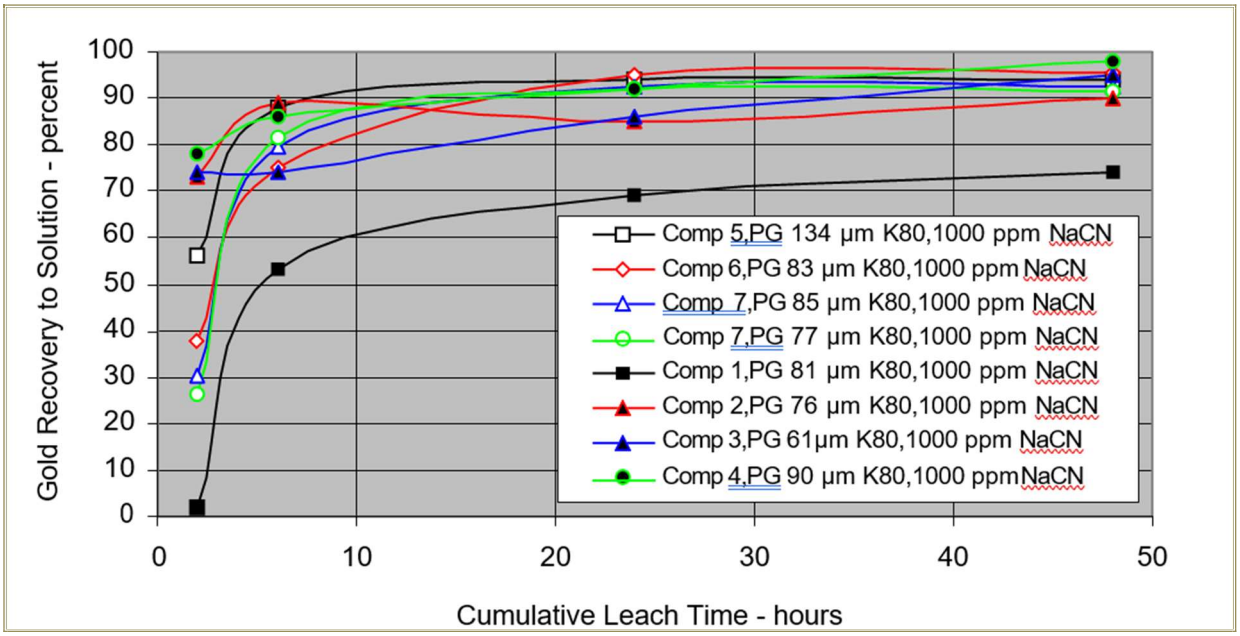
Gravity and leach tests were completed on the Master composite and eight variability composites. The gold extraction ranged between 91% and 99%, with an average gravity recovery of 43%, and an overall average extraction of 95% using varying grind sizes. The average silver recovery for all the composites to the gravity concentrate was 13%, and overall extraction 86.5%. Leach testwork without gravity concentration had an average overall extraction of 90%, with higher cyanide consumptions. The flowsheet with gravity concentration is illustrated in Figure 13-3.



Source: G&T KM2121 (2008)

Figure 13-3: Leach Flowsheet

The curves shown in Figure 13-4 indicate fast leach kinetics at 1,000 ppm cyanide and varying grind sizes, with most of the gold extracted within 24 hours, and leaching complete by 48 hours. Low cyanide and lime consumptions were reported (Table 13-5).



Source: G&T KM2121 (2008)

Figure 13-4: Gravity/Leach Curves

Table 13-5: Summary of Gravity and Leach Tests

Test	Composite	Consumption		Primary Grind (P_{80}, µm)	Feed Grade (Au g/t)	Gravity Recovery (%)	48 h Extraction (%)	Overall Recovery (%)
		NaCN	Lime					
35	1	0.1	0.5	82	6.1	65	74	91
36	2	0.3	1.1	76	7.5	39	90	94
37	3	0.3	0.8	61	4.6	53	95	98
38	4	0.4	0.3	90	9.6	52	98	99
20	5	0.8	1.1	138	14.6	30	94	96
21	6	1.5	1.2	83	13.3	27	96	97
22	7	1.1	1.6	85	21.8	36	93	95
23	8	0.8	1.5	77	47	39	91	95
Average	-	0.7	1	87	15.6	43	91	95

Source: G&T KM2121 (2008)

13.6 G&T KM2522 (2010)

The metallurgical drill core collected for the 2008 program and composite material from KM2121 was used to prepare low- and high-grade composites to reflect the forecast open pit head grades of 5 g/t Au and 15 g/t Au from underground mining. Silver head grades of 13 g/t and 24 g/t were reported for the low- and high-grade composites, respectively. Silver assays were not included as part of the test program.

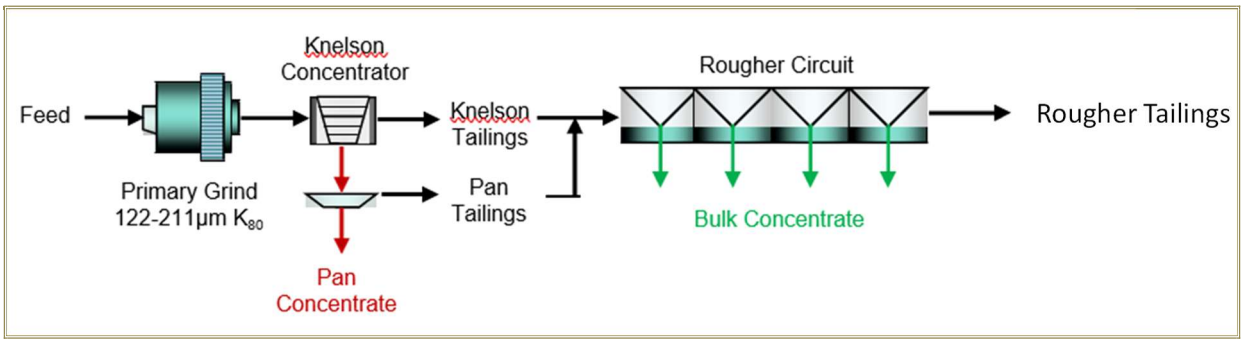
BWi testwork was completed on both the high- and low-grade composites at a closing screen size of 106 µm. The low-grade composite measured 13.4 kWht/t, and the high-grade 14.1 kWh/t, indicating that the material is of medium hardness.

Gravity tests were completed on both composites at varying grind sizes, ranging from P_{80} 122 µm to P_{80} 266 µm. The gravity recovery ranged from 28% to 65%.

Three flowsheets were investigated in the KM2522 test program. The first series of tests included a primary grind of P_{80} 122 µm with gravity concentration, followed by leach of the gravity tailings; the second flowsheet included gravity and rougher flotation at a primary grind P_{80} of 122 µm to 211 µm; and the third set of tests included gravity, rougher, and cleaner flotation, with leach of the cleaner concentrate at an initial grind size of P_{80} of 122 to 211 µm.

The gravity leach testwork included four tests: one on the low-grade composite, and three on the high-grade composite. The results indicated that approximately 96.3% of the gold can be extracted using gravity followed by 48-hour leach for the low-grade composite at P_{80} 122 µm. The high-grade composite, with primary grind sizes of P_{80} 123 µm, P_{80} 99 µm, and P_{80} 64 µm, had gold leach extractions of 90.0%, 91.1%, and 93.6%, respectively. The leach kinetics for all four tests were similar, with most of the gold extracted by 24 hours. The reagent consumption for the low-grade material was low, while the high-grade leach had higher cyanide and lime consumptions.

The second set of tests were carried out using a gravity and rougher flotation flowsheet. The flowsheet is shown in Figure 13-5.



Source: G&T KM2522 (2010)

Figure 13-5: Gravity—Rougher Flotation Flowsheet

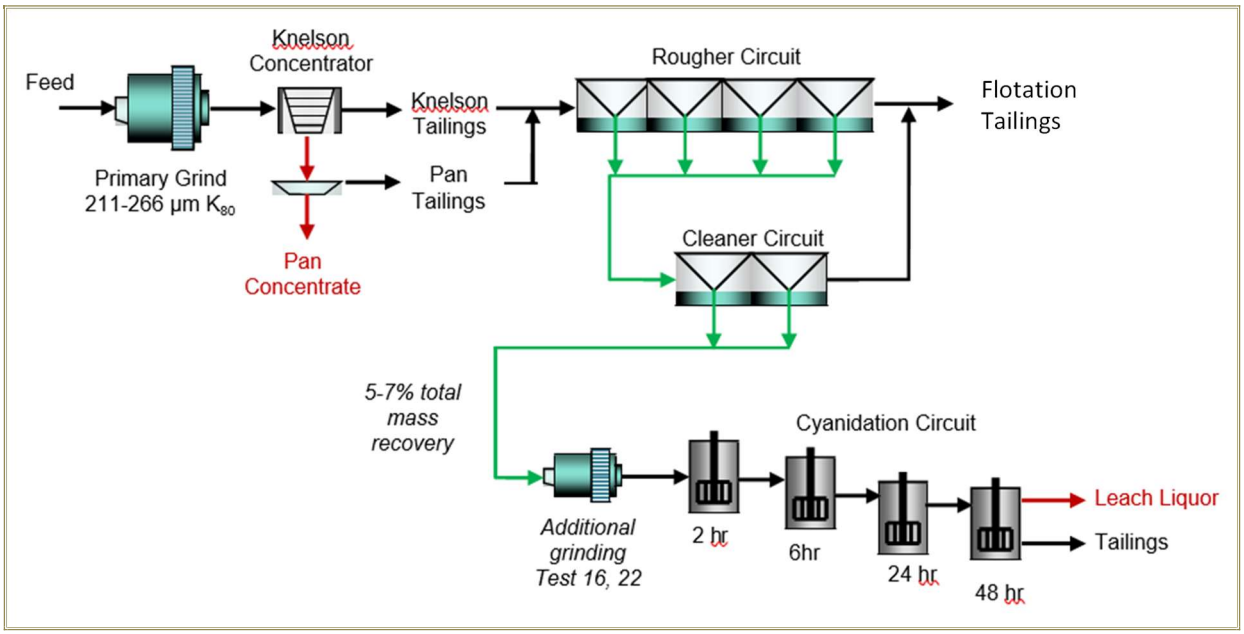
The results indicate that 97.7% to 98.8% of the gold, with combined grades of 80 g/t to 182 g/t, can be recovered to the gravity pan and rougher concentrates at grind sizes of approximately P_{80} 122 µm and 211 µm. The gravity recovery ranged from 39.3% to 68.1%, and the staged rougher recovery varied from 94.0% to 98.0%. The total mass recovery of gravity and rougher concentrate averaged 6.4% for the low-grade composite, and 9.8% for the high-grade composite. The tests were completed using an 8-minute residence time, with PAX and MIBC to aid in flotation. The results are summarized below in Table 13-6.

Table 13-6: Gravity/Rougher Tests

Sample	Test #	Grind Size (P_{80}, µm)	Total Mass Rec. (%)	Gold Recovery (%)		Total (%)	Concentrate Grade (g/t Au)			Au Grade (g/t)	
				Pan Conc.	Float. Conc.		Pan	Rougher	Combined	Feed	Rougher Tailings
Low-Grade	3	122	6.4	46.0	51.7	97.7	244	55	87	5.68	0.14
Low-Grade	11	211	6.3	68.1	30.0	98.2	146	39	80	5.11	0.10
High-Grade	4	123	10.2	44.1	54.7	98.8	938	92	154	15.9	0.21
High-Grade	8	208	9.4	39.3	59.5	98.8	593	125	182	17.2	0.23

Source: G&T KM2522 (2010)

The third series of tests included three flotation leach tests: one using the low-grade composite, and two using the high-grade composite. The flotation tests were carried out at a coarse grind P_{80} of between 211 µm and 266 µm, with a regrind of the cleaner concentrates prior to leach on both composites. The total gold recoveries for the circuit, which included gravity concentration, flotation of the gravity tailings, and cyanidation leaching of the reground flotation concentrate, were 95% and 94% for the low-grade and high-grade composites, respectively. The flowsheet, summary of results, and leach curves are shown in Figure 13-6, Table 13-7, and Figure 13-7, respectively.



Source: G&T KM2522 (2010)

Figure 13-6: Gravity/Flotation/Leach Flowsheet

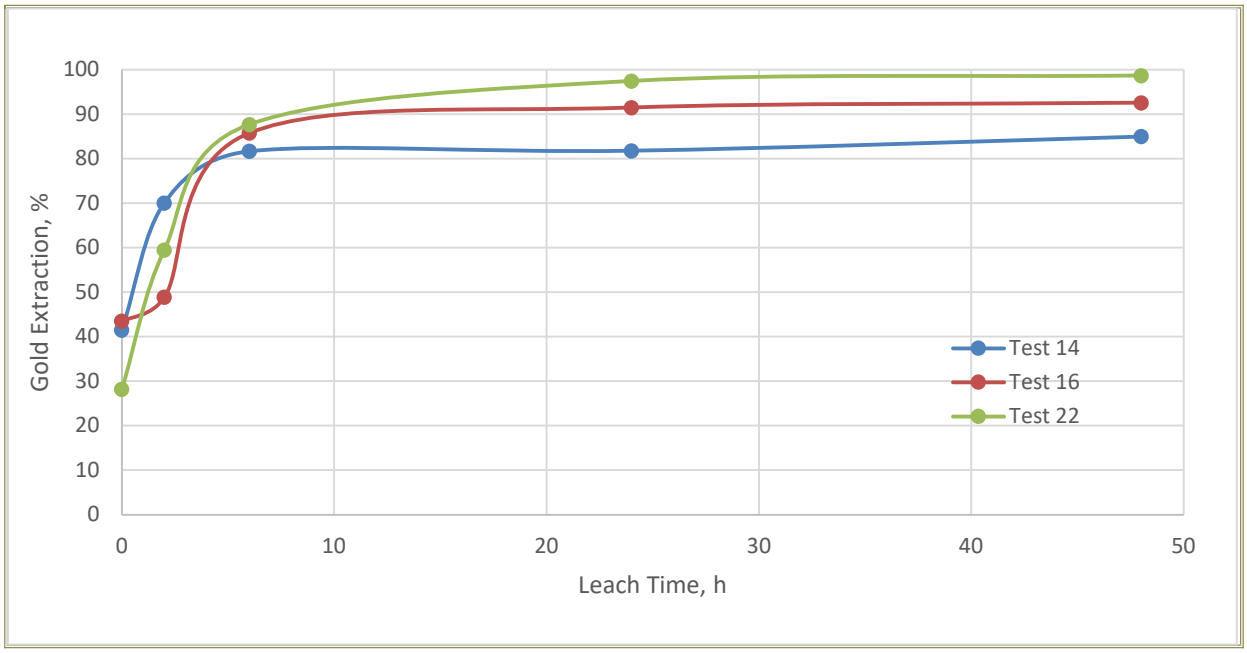
The flotation circuit included an 8-minute rougher residence time and 10-minute cleaner residence time. The staged recovery to the cleaner concentrate was approximately 95% for both composites, with mass pulls in the range of 4.5% to 7.5%. Regrind of the cleaner concentrate resulted in better leach

performance compared to the high-grade composite with no regrind. For the high-grade composite, overall recovery with gravity was 89% for the coarse grind, and 94% with regrind of the leach feed.

Table 13-7: Summary of Concentrate Leach Results

Composite	Leach Test	Grav. Float. Test	Grav. Au Recovery (%)	Float. Au Recovery (%)	Float. Au Stage Recovery (%)	Mass Recovery (%)	Leach Feed		Leach Au Extraction (%)	Total Au Recovery (%)	Cyanide Consumption (g/t of feed)
							(P_{80} µm)	Au Grade (g/t)			
Low Grade	22	19	28.2	67.8	94.4	4.5	18	77	98.7	95.1	0.32
High Grade	14	13	41.5	55.9	95.6	6.5	252	135	85.0	89.0	1.16
	16	15	43.5	54.5	96.5	7.5	53	129	92.6	94.0	0.5

Source: G&T KM2522 (2010)



Source: G&T KM2522 (2010)

Figure 13-7: Gold Extraction vs. Leach Time

A comparison of the high-grade leach tests T14 and T16 indicates that regrind of the cleaner flotation concentrate improved extraction. Higher cyanide consumptions were noted at the finer grinds.

13.7 ALS KM4263 (2014)

The material for the testwork was collected from the 1300 vein. The program included gravity concentration followed by rougher flotation flowsheet and cyanide leach of the rougher tailings. The tests included four rougher stages for a total residence time of 8 minutes using reagents PAX and MIBC to aid

in flotation. The head assays of Composite 1 can be found in Table 13-8 and testwork summary details and results are listed in Table 13-9.

Table 13-8: Chemical Composition of Composite 1

Sample	Assays			
	Au (%)	Ag (%)	Fe (g/t)	S (g/t)
Composite 1 HG 1	86.5	78	7.7	8.3
Composite 1 HG 2	93.0	88	8.1	8.6
Average	89.7	83	7.9	8.4

Source: ALS KM4263 (2014)

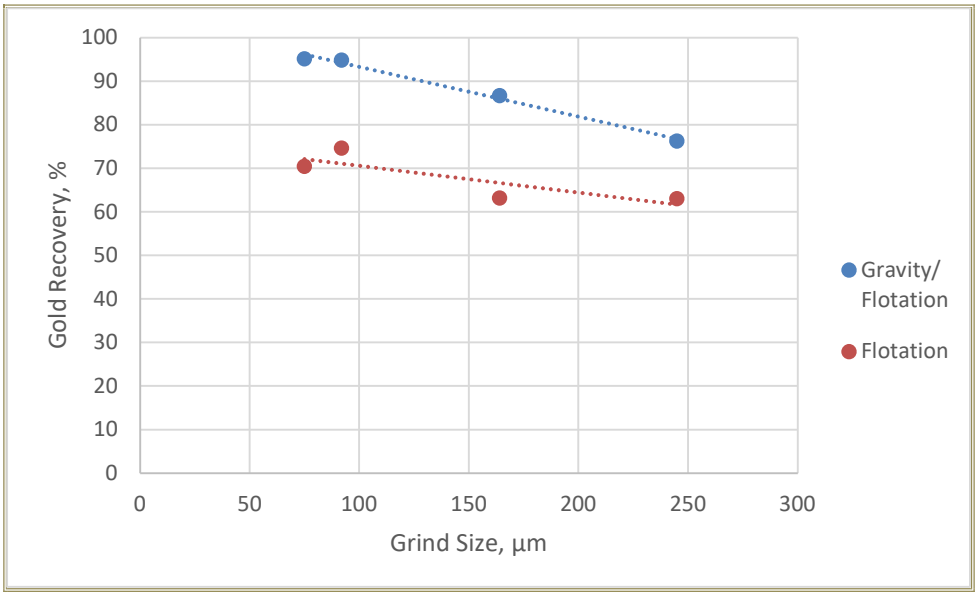
Table 13-9: KM4263 Flotation Test Result Summary

Test Number	Au Head Grade (g/t)	Grind Size P_{80} (µm)	Rougher Mass Pull (%)	Gravity Recovery (%)	Overall Recovery (%)	Gold Concentrate Grade (g/t)
4263-01R	93.4	245	21.1	-	63.1	280
4263-02GR	83.6	245	16.9	41.1	76.3	348
4263-03GR	88.8	164	22.8	51.3	86.7	337
4263-04R	93.0	164	19.5	-	63.2	301
4263-05GR	97.3	92	25.3	38.9	63.7	245
4263-06R	76	92	21.4	60.6	74.7	266
4263-07GR[1]	92.9	92	27.8	68.3	94.9	317
4263-08GR[2]	84.5	75	30.1	66.1	95.2	267
4263-09R	87	75	22.1	-	70.5	276

Source: ALS KM4263 (2014)

Notes: [1] Repeat of Test 5 with more collector
[2] Grind Size Estimated based on grind time compared to Test 7.

The best results were achieved with gravity at a grind size of 75 µm and a higher collector dose in the rougher flotation stage. Minor element analysis was completed on test 4263-07GR and the results showed levels of Arsenic and Uranium that may encounter smelter penalties. Figure 13-8 illustrates the relationship between grind size and recovery.

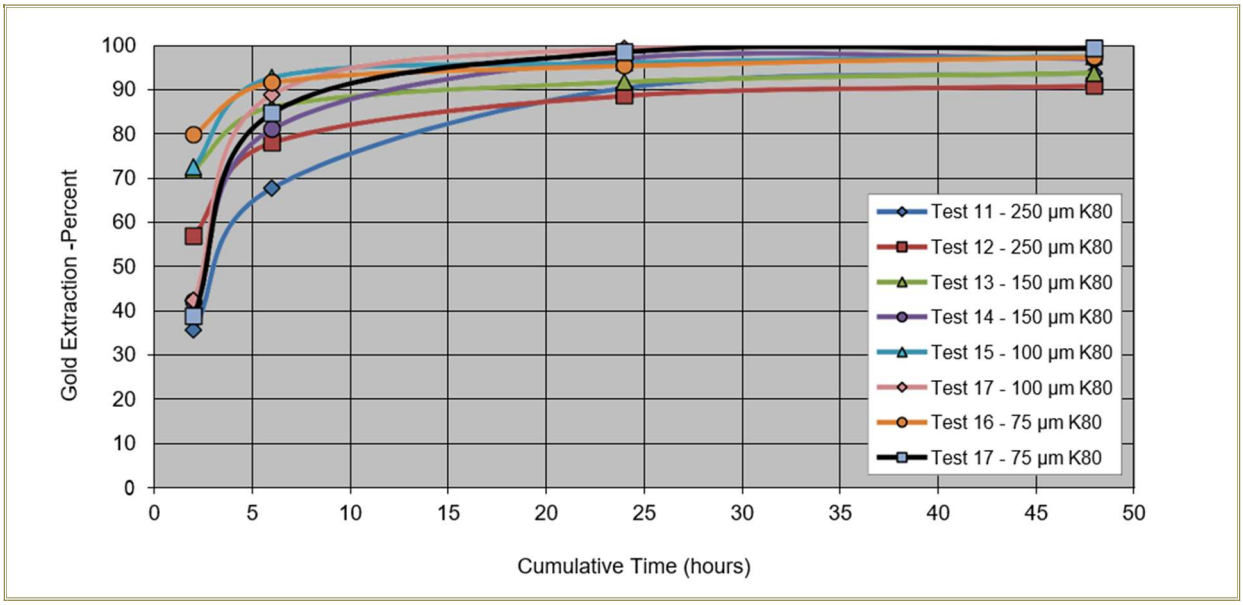


Source: K-Met (2020)

Figure 13-8: Grind Size vs. Gold Recovery

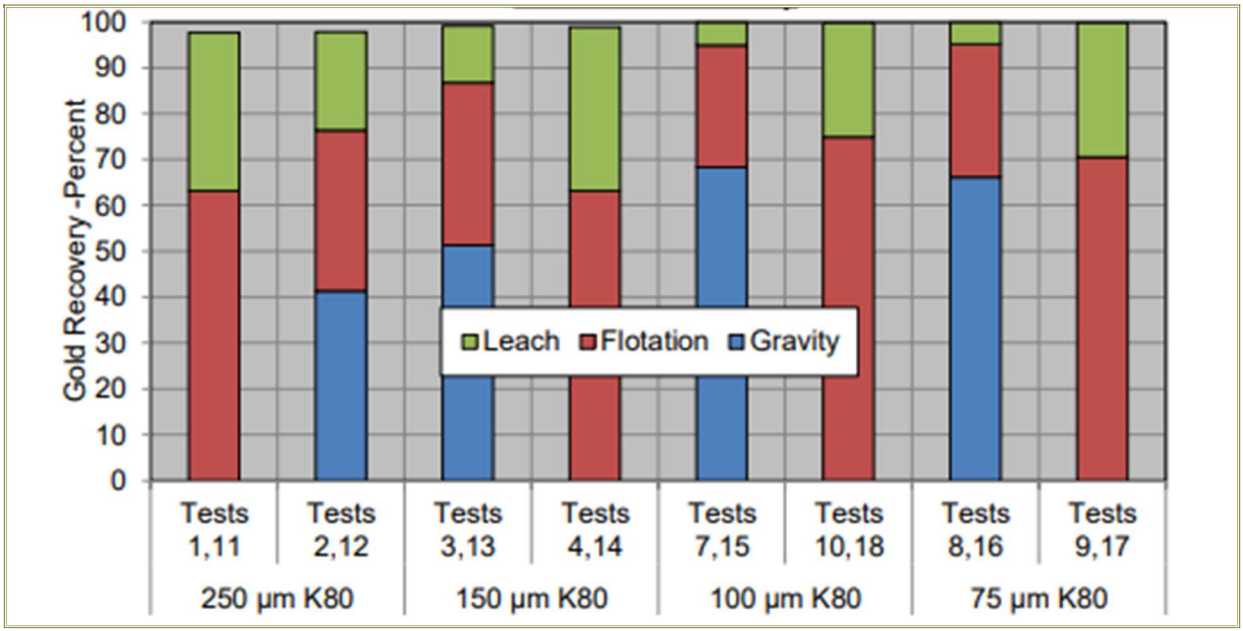
The effect of primary grind size on gold extraction was investigated using a 48-hour leach at a cyanide concentration of 1,000 ppm on the rougher tailings from the previous tests. The gold extraction ranged from 91% to 99% and averaged 96%. Silver ranged from 75% to 96% and averaged 86%. The majority of the gold was extracted in 24 hours.

The results indicate the total gold recovery from all circuits ranged from 97.7% to 99.9% for all grind sizes (75 µm, 100 µm, 150 µm, and 250 µm). Silver recovery ranged from 92.9% to 99.7%. Cyanide consumption increased with finer grind size. The results at a primary grind of P_{80} 75 µm and 100 µm had similar results with and without gravity followed by rougher flotation and cyanide leach of the rougher tailings. The gold extraction leach curves are illustrated in Figure 13-9. The results for overall gold and silver recovery are shown in Figure 13-9, and Figure 13-10, respectively.



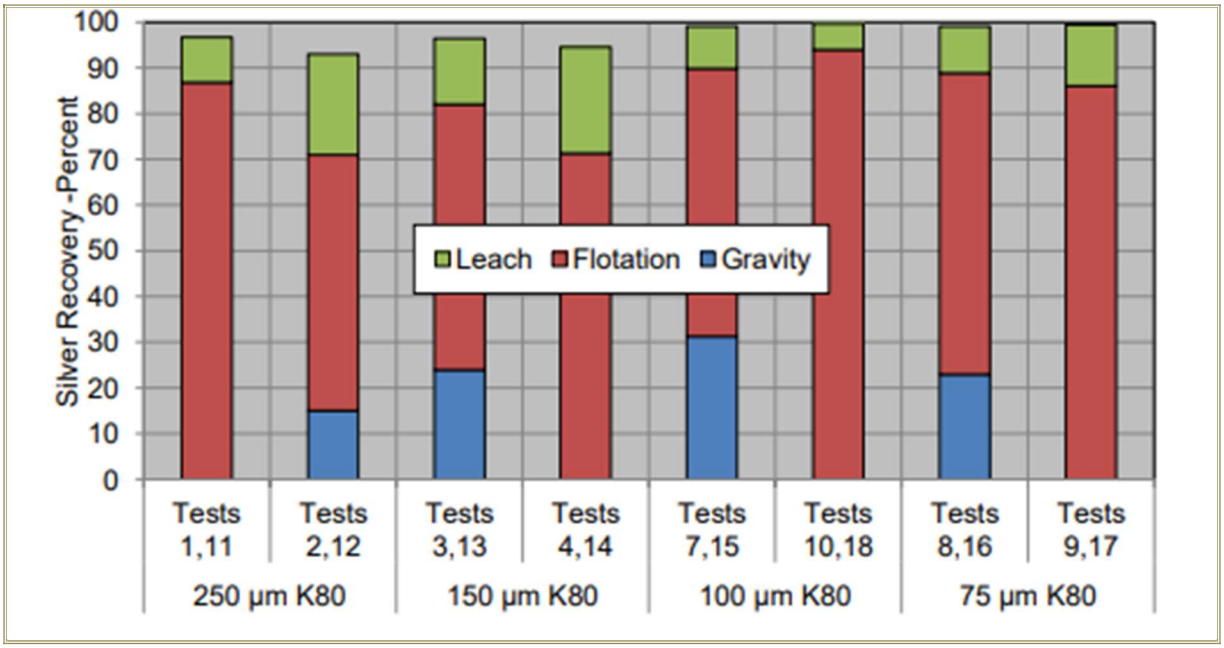
Source: ALS KM4362 (2014)

Figure 13-9: Gold Extraction vs. Time



Source: ALS KM4362 (2014)

Figure 13-9: Total Gold Recovery



Source: ALS KM4362 (2014)

Figure 13-10: Total Silver Recovery

13.8 Summary of Results

Three options were investigated for processing the Elk Project material in the range of the anticipated gold feed grades: Whole-ore Leach (WOL), flotation, and leach of a cleaner concentrate. All three options included gravity concentration. With all flowsheets, gold recovery in the low- to mid-90 percentile was achieved. The testwork completed shows the flotation with gravity can produce saleable rougher and cleaner concentrates. The whole-mineralized material leach circuit with gravity concentration at a primary grind in the range of P_{80} 106 µm achieved an average gold extraction of 95%.

The first option includes gravity concentration followed by whole-ore leach of the gravity tailings to produce gold doré. The testwork carried out on Composites 1 through 6 and the low-grade and high-grade composites from Programs KM2121 and KM2522 were used to evaluate the gravity/leach circuit. The tests were completed at primary grind sizes ranging from P_{80} 61 µm to 138 µm, at a cyanide concentration of 1,000 ppm, for 48 hours leach time. The average head grade of the samples tested was 10.5 g/t, with a gravity recovery of 39.5%, and an overall gold extraction of 95%. Consumption rates averaged 0.6 g/t for cyanide, and 1.0 g/t for lime. Silver assays were reported only in test Program KM2121 and the average overall extraction of all composites tested was 86.5%.

The second flowsheet included gravity concentration and flotation to produce a saleable gold concentrate, and was based on a review of testwork completed on the low-grade and Master composites. The average rougher gold recovery for all tests, based on an 8-minute laboratory residence time, was 94.8%, with a grade of 50 g/t Au, and an average mass pull of 8.5%. The three cleaner tests indicate that

a staged recovery of 93.8% with a grade in the range of 79 g/t can be achieved after 10 minutes of flotation residence time, with a mass recovery in the range of 5% to 7%. The flotation reagent consumption rates were approximately 120 g/t PAX and 30 g/t MIBC. The results are shown below in Table 13-10.

Silver assays were not included in the flotation elements that were measured except in test Program KM4263 completed on a high-grade composite, 83 g/t Ag. The silver recovery to the rougher concentrate ranged from 70.9% to 93.9% at grind sizes between P_{80} 75 µm and 250 µm. The highest recoveries were at a grind size of P_{80} 100 µm.

Table 13-10: Gravity/Flotation Results

Sample	Test #	Head Grade (g/t Au)	Grind Size (P_{80}, µm)	Mass Pull (%)			Concentrate Grade (g/t Au)			Concentrate Recovery (% Au)				
				Pan	Rougher	Cleaner	Pan	Rougher	Cleaner	Overall	Pan	Rougher	Cleaner	Overall
Low-Grade	KM2522-11	5.11	211	2.4	2.7	-	146	55	-	80	68.1	91.9	-	97.4
Low-Grade	KM2522-03	5.68	122	1.1	5.3	-	244	55	-	87	46.0	95.7	-	97.7
Low-Grade	KM2522-19	5.08	211/18	0.4	6.6	4.5	397	52	77	103	28.2	95.1	94.4	96.0
Master Composite	KM2121-39	10.7	102	1.6	14.7	5.7	310	38	71	114	47.0	96.2	94.8	94.8
Master Composite	KM2121-40	12	102	1.7	13.2	7.3	281	51	89	125	40.7	95.1	92.1	95.3
Average	-	8.4	102 to 211	1.4	8.5	5.8	276	50	79	114[1]	46.0	94.8	93.8	95.4[1]

Source: G&T KM2121, KM2522 (2008, 2010)
Notes: [1]Average of cleaner flotation tests only

The third option considered was gravity concentration, followed by flotation of the gravity tailings with a single stage of cleaning, with or without regrind of the cleaner concentrate, and a 48-hour leach. The one low-grade test achieved a high gold leach extraction of 98.7% with a fine regrind to a P_{80} of 18 µm. The cleaner recovery to the leach for the low-grade composite was 96% with a grade of 77 g/t Au at a coarse primary grind of 211 µm. Leach of the high-grade cleaner concentrate at a primary grind P_{80} of 266 µm resulted in overall gold extractions of 88.8% without regrind, and 94.0% with regrind to a P_{80} of 57 µm. The tests with regrind required high cyanide dosages to achieve the higher gold extractions.

Very few tests were completed in the range of the gold head grades reported in the mine plan. Additional testwork is recommended with variability composites and composites that represent the mine plan for the first few years of operation to confirm the flowsheet and optimize the design parameters. Sulphur and gold should be investigated in the next phase of testwork, since there appears to be a correlation between the Au:S ratio and the final concentrate grade produced with a gravity/flotation flowsheet.

14 MINERAL RESOURCE ESTIMATE

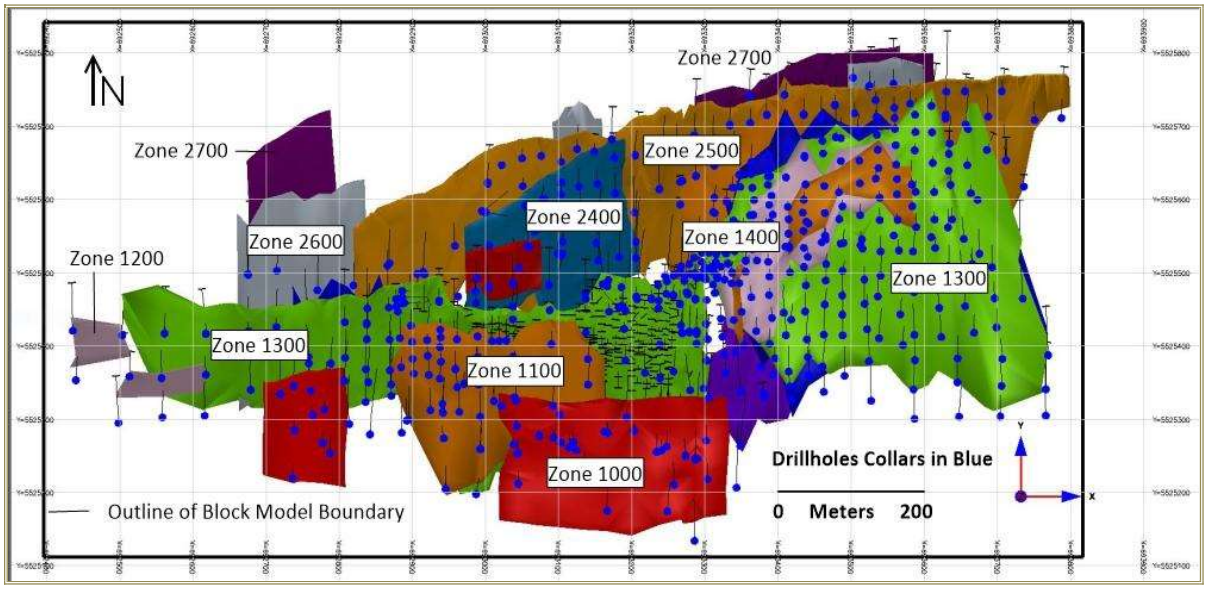
GMMC provided GMRS with drill-hole collar, down-hole survey, and assay data in Excel format, and wireframe models of 14 mineralized zones and a surface topographic model, all in DXF format. GMMC also provided copies of previous technical reports on the Property.

14.1 Exploratory Data Analysis

The drill-hole collar file contained location data for 1,053 drill holes, of which 933 intersected the 14 modelled mineralized zones. The assay file contained 17,253 gold and silver assays, of which 4,204 were contained within the modelled zones. The number of composites exceeds the number of assays because the compositing process incorporated unsampled intervals into the composite population. Table 14-1 shows the number of assays and composites within each zone. Figure 14-3 and Figure 14-4 show the drill holes and mineral zones in plan view.

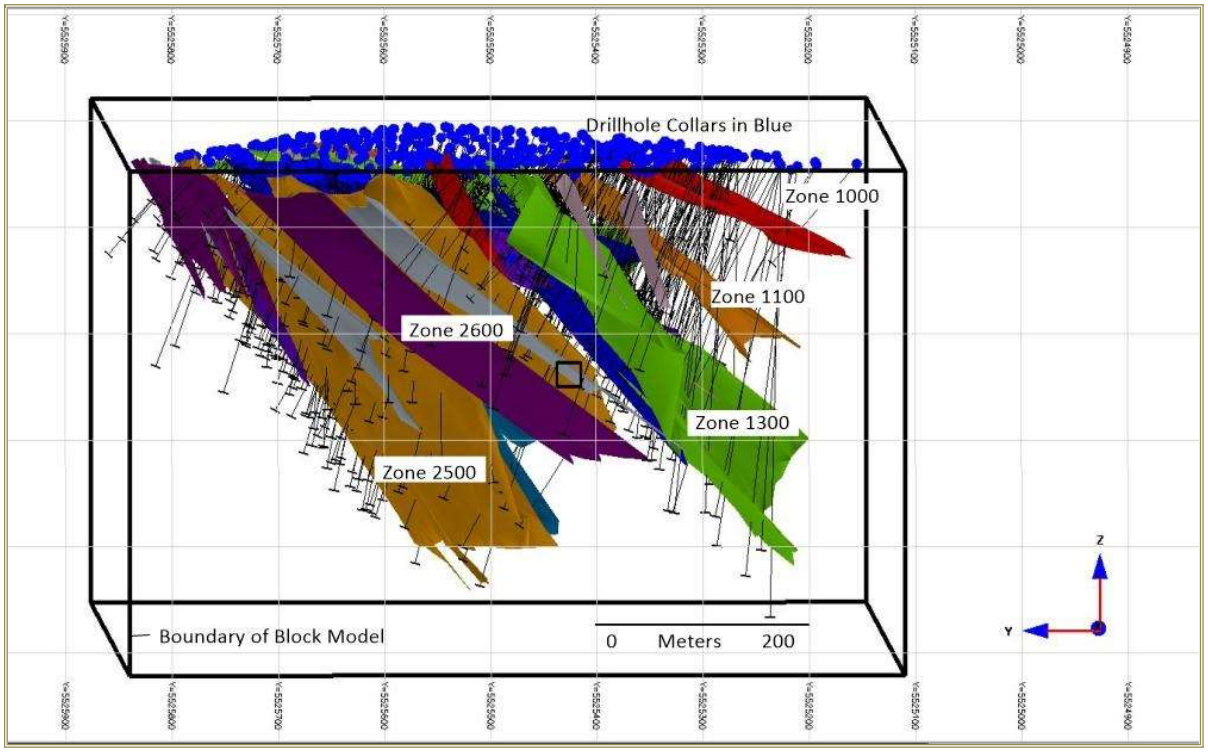
Table 14-1: Elk Gold Drill Hole and Assay Statistics by Zone

Zone	Assays	Composites
Vein 1000	48	121
Vein 1100	132	270
Vein 1200	134	251
Vein 1300	2,594	5,007
Vein 1400	492	866
Vein 1500	77	125
Vein 1700	27	63
Vein 2200	33	67
Vein 2400	63	113
Vein 2450	6	9
Vein 2500	487	759
Vein 2500	8	24
Vein 2600	49	118
Vein 2700	54	97
Total	**4,204**	**7,890**



Source: GMRS, August 2020

Figure 14-1: Elk Gold Drill Holes and Mineralized Zones Plan View



Source: GMRS, August 2020

Figure 14-2: Elk Gold Drill Holes and Mineralized Zones Perspective View Looking East

The property was tested by drilling between 1989 and 2012, using a variety of core and sample sizes. Table 14-2 shows the mean gold grade of assays by year. All average grades are similar, except for those samples collected during 1994 and 1995, when exploration focused on underground drill testing of Vein 1300, the largest of the known zones. Because the 1994 and 1995 samples were selective in the target and area of that target that was tested, their elevated means, relative to samples from other years, does not disqualify them from use in the resource estimate that follows. All assay values that are contained within the 14 modelled zones of mineralization were considered suitable for the purpose of resource estimation.

Table 14-2: Elk Gold Average Gold Grade by Year

Year	Count	Minimum	Maximum	Mean
1989	312	0	189.69	1.79
1990	1,244	0	273.40	2.09
1991	599	0	916.74	3.71
1994	448	0	548.71	10.47
1995	1,520	0	1,176.18	10.67
1996	1,222	0	196.12	2.50
2000	216	0	91.06	2.80
2002	514	0	103.84	2.02
2003	693	0	315.84	2.41
2004	881	0	122.68	1.98
2005	993	0	135.97	1.76
2006	1,190	0	120.86	1.41
2007	100	0	9.90	0.53
2010	3,666	0	432.00	1.03
2011	2,734	0	465.00	1.34
2012	4,069	0	220.00	0.58

14.2 Capping

Capping is the process of artificially reducing high values within a sample population that are regarded as statistically anomalous with respect to the population as a whole (outliers), to avoid the distorting influence these values would have on the statistical characteristics of the population if left at their full value. The risk of including atypically high values in a Mineral Resource Estimate is that their contribution to the estimated grade will be disproportionate to their contribution to the tonnage, and therefore the grade of the resource as a whole will be overstated.

Cumulative frequency curves are commonly used to determine whether capping is appropriate. If a single sample population is present, the cumulative frequency curve is a relatively straight line; steps in the curve indicate the potential presence of separate or mixed populations.

Figure 14-3 shows the cumulative frequency curve for gold for all Elk Gold Project drill core assays. There is a noticeable break in the curve at approximately 300 g/t, and that was taken as the capping value. There are 11 assay values greater than 300 g/t and, if they are capped at 300 g/t, the aggregate value of the assay population drops by 4%. This is not a large reduction, and indicates that exceptionally high gold values are not common.

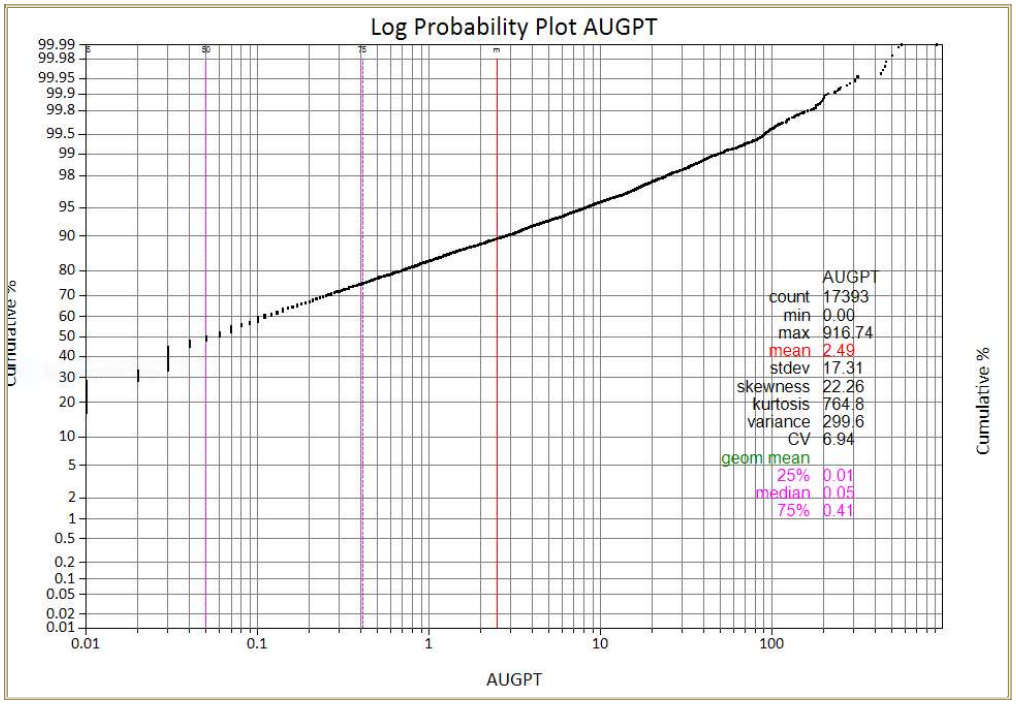


Figure 14-3: Elk Gold Cumulative Frequency Curve for Gold Assays

Figure 14-4 is the cumulative frequency plot for silver assays. The trend has a break at 600 ppm, but only four assays fall above that break and their influence on the total population is considered to be negligible; consequently, no silver assays were capped.

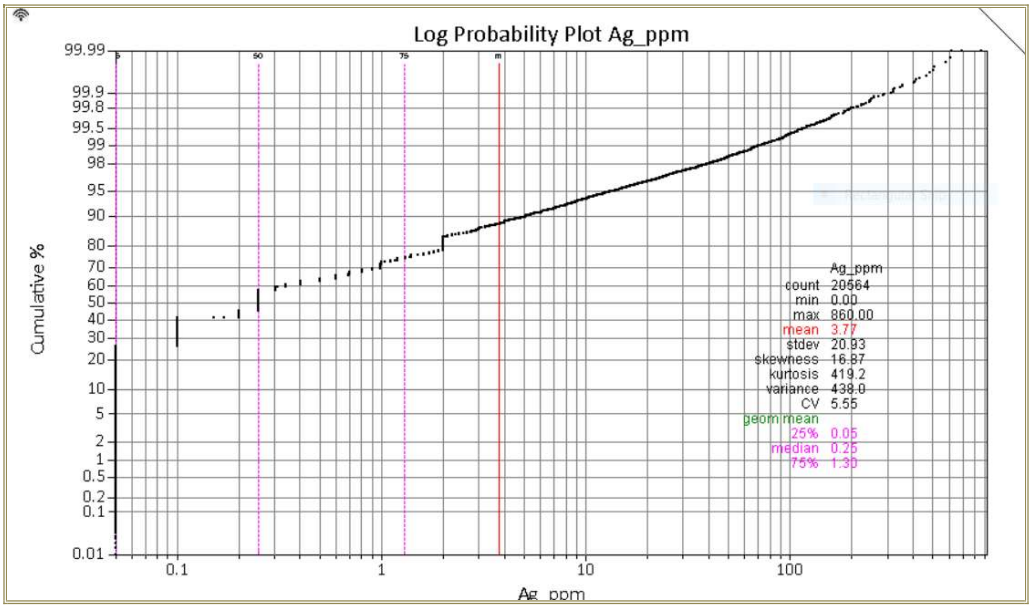


Figure 14-4: Elk Gold Cumulative Frequency Curve for Silver Assays

14.3 Composites

Compositing of samples is done to overcome the influence of sample length on the contribution of sample grade (sample support). There is a wide range of sample lengths, from a maximum of 4.33 m to a minimum of 0.03 m, but 0.5 m is both the mode and median length, so that length was chosen as representative, and all samples were composited to 0.5 m. Table 14-1 shows the number of composites in each of the 14 zones. As approximately 57% of the samples were greater in length than 0.5 m, the compositing process created more composites than the original samples; in addition, unsampled intervals were incorporated into the composite population.

14.4 Bulk Density

Files pertaining to historical work on the Property contain records for 125 bulk-density measurements. Seventy-three were made in 1995 and another 52 were made at an unknown date; all were made using the Archimedes method of immersion in water. Measured values ranged from a high of 4.11 g/cm^3 to a low of 2.42 g/cm^3. The highest value contained more than 50% pyrite. Descriptions of samples with low values indicate that these samples were fractured. Thus, neither end of the spectrum of values is representative of the deposit as a whole. Most of the rock within which the deposit is contained is of granitic composition, for which a bulk density of 2.73 g/cm^3 is a common value for granitic rock. A density of 2.7 g/cm^3 was used in the current Mineral Resource estimate.

14.5 Geological Interpretation

GMMC provided GMRS with wireframe models of 14 mineralized zones that were used as the basis of a previous historical resource estimate in 2016. These wireframes represent the best current interpretation of the distribution of mineralization within the Property. Two of the zones (Veins 1300 and 2500) represent approximately 55% of the total volume of all 14 zones. Several others—1700, 2200, 2450, and 2550—collectively represent less than 4% of the total volume of all zones. Several of the wireframes, as received projected above the current topographic surface, and were truncated at that topographic surface.

Table 14-3 lists the volume of each zone in cubic metres, and the percentage of the total volume represented by each zone. Table 14-3 also shows the number of assays that support the resource estimation within each zone. Vein 1300 contains approximately 62% of the total number of assays used, largely resulting from the underground drilling. This high quantity of data means that a higher level of confidence can be placed in the resultant estimate for Zone 1300 than for the other zones.

Table 14-3: Elk Gold Modelled Mineralized Zones

Zone	Volume (m^3)	Percent of Volume	Assays	Percent of Total
V1000	87,771	4	48	1
V1100	109,507	5	132	3
V1200	66,450	3	134	3
V1300	812,745	34	2594	62
V1400	316,432	13	492	12
V1500	74,434	3	77	2
V1700	21,188	1	27	1
V2200	30,465	1	33	1
V2400	88,955	4	63	1
V2450	6,788	0	6	0
V2500	517,500	22	487	12
V2550	8,906	0	8	0
V2600	126,430	5	49	1
V2700	131,021	5	54	1
Total	**2,398,592**	**100**	**4,204**	**100**

14.6 Analysis of Spatial Continuity

Variographic analysis using Sage 2001™ software was carried out on non-zero gold assays for Zones 1300, 1400, and 2500, for which it was considered that there were sufficient samples to produce meaningful variograms. The results are tabulated in Table 14-4. Because of similarity of orientation, variographic

values for Zone 1300 were used for Zones 1000, 1200, 1500, and 2200. Likewise, variographic values for Zone 2500 were used for Zones 1700, 2200, 2400, 2450, 2550, 2600, and 2700.

The same variographic values were used for silver.

Table 14-4: Elk Gold Variography Zones 1300, 1400, and 2500

Name	Type	Sill	Range			Azimuth (°)	Dip (°)	Spin (°)
			Long	Median	Short			
V1300_Au	Nugget	0.59	0	0	0	0	0	0
V1300_Au	Spherical	0.37	25	10	4	38	-66	0
V1300_Au	Spherical	0.05	1,000	535	38	180	72	0
V1400_Au	Nugget	0.28	0	0	0	0	0	0
V1400_Au	Spherical	0.65	220	29	3	223	35	0
V1400_Au	Spherical	0.07	1,000	147	2	302	61	0
V2500_Au	Nugget	0.82	0	0	0	0	0	0
V2500_Au	Spherical	0.15	464	210	2	113	12	0
V2500_Au	Spherical	0.03	749	249	34	283	45	0

Search ellipses were constructed for each of the zones based on drill hole spacing and orientation of the zone. Search ellipse parameters are shown in Table 14-5. Because their strikes and dips are variable, dynamic ellipses were used for Zones 1100, 1200, and 1300.

Table 14-5: Elk Gold Search Ellipses

Name	Azimuth (°)	Dip (°)	Spin (°)	Major (m)	Median (m)	Minor (m)
V2700	160	-55	0	50	50	75
V2600	170	-55	0	60	40	20
V2550	170	-60	0	35	50	20
V2500	80	0	45	75	50	50
V2450	170	-55	0	50	25	10
V2400	170	-45	0	75	50	25
V2200	170	-20	0	50	40	25
V1700	170	-60	0	50	35	25
V1500	80	0	30	50	75	25
V1400	90	0	30	75	50	50
V1200	85	0	30	75	50	25
V1300	90	0	45	75	50	50
V1100	85	0	30	75	50	50
V1000	85	0	30	75	50	50

14.7 Block Model

Block model parameters are given in Table 14-6. The origin is the block centroid for minimum X, Y, and Z. Each block was discretized 10 x 5 x 5 (x, y, z directions). The model was not rotated.

Table 14-6: Elk Gold Block Model Parameters

	Origin (m)	Size (m)	Number	
X	692,405	10	Columns	142
Y	5,525,117.5	5	Rows	146
Z	1,182.5	5	Levels	95

14.8 Interpolation Plan

Gold and silver grades were interpolated into the block model in a single pass. For a grade to be interpolated into a block it was necessary that a minimum of two and a maximum of eight composites be located within the volume of the search ellipse. A maximum of one composite per drill hole was allowed, to ensure that geological continuity was demonstrated by requiring that each block was informed by a minimum of two drill holes.

14.9 Metal Equivalency

The current mineral resource estimate includes both gold and silver assay values and the combined value is expressed as a gold equivalency. The formula by which the two metals are combined is:

AuEq = ((Au_Cap*41.8*0.96) + (Ag_Cap*0.55*0.86))/(41.8*0.96)

The values of the relevant parameters are set out in Table 14-7.

Table 14-7: Elk Gold Metal Equivalency Parameters

Parameter	Unit	Value
Gold Grade	g/t	Variable
Silver Grade	g/t	Variable
Gold Price	US$/oz	1300.00
Gold Price	US$/g	41.80
Silver Price	US$/oz	17.00
Silver Price	US$/g	0.55
Recovery of Gold	%	0.96
Recovery of Silver	%	0.86

14.10 Mineral Resource Classification

Mineral resources were classified as Measured, Indicated, or Inferred according to the criteria set out in Table 14-8.

Table 14-8: Elk Gold Resource Classification Criteria

Classification on Category	Composites			Azimuth (°)	Dip (°)	Spin (°)	Major (m)	Median (m)	Minor (m)
	Min.	Max.	Max./Holes						
Measured	8	8	1	0	0	0	25	25	25
Indicated	4	8	1	0	0	0	50	50	50
Inferred	2	8	1	0	0	0	75	75	75

The Measured category was restricted to Zone 1300 because that is the only zone that has been explored by underground development and drilling that has provided 3-D exposures of the mineralization as well as close-spaced testing by underground drilling. Resources in all the other zones were classified as Indicated or Inferred.

14.11 Reasonable Prospects of Eventual Economic Extraction

Because the Elk Gold Mineralization occurs in part at or near surface, it is necessary to demonstrate the potential economic viability of the near-surface portion of the resource by constraining the block model with a conceptual pit. Parameters for the pit are set out in Table 14-9. The economic contribution of silver was not incorporated into the design of the conceptual pit as its potential impact is not considered to be material.

Table 14-9: Elk Gold Conceptual Pit Parameters

Parameter	Unit	Value
Gold	US$/oz	1,361
Gold	US$/g	43.76
Exchange Rate	US$:C$	0.74
Production Rate	t/d	900
Mining Cost	$/t material	2.70
Processing + G&A + Trucking Costs	$/t ore	26.00
Recovery Au	%	92
NSR	%	2
Selling Cost	%	2
Pit Slope	degrees	50

14.12 Mineral Resources Tabulation

The Mineral Resource is shown in Table 14-10 for a range of cut-off grades of capped gold. The resource is divided into pit-constrained and underground portions. The pit-constrained resource is stated at a base-case cut-off grade of 0.5 g/t Au, and the underground resource is stated at a base-case cut-off grade of 5.0 g/t Au (Table 14-11). Ounces have been rounded to the nearest hundred and tonnes have been rounded to the nearest thousand. The base case for both pit-constrained and underground resources is highlighted.

Table 14-10: Elk Gold Pit-Constrained Resource Tabulation

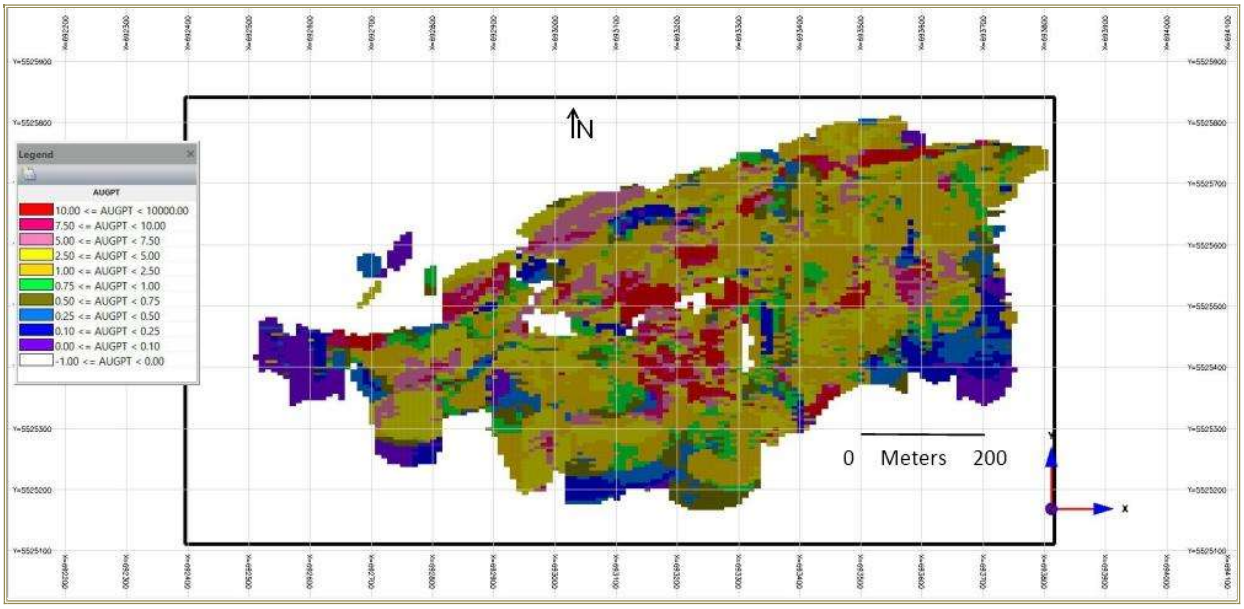
Cut-off Au Eq 5.0 g/t	Tonnes	AuEq (g/t)	Au Capped (g/t)		AuEq Ounces
Measured	141,000	12.9	12.8	11.8	58,000
Indicated	681,000	10.1	9.9	17.4	217,000
Measured + Indicated	821,000	10.6	10.4	16.5	275,000
Inferred	72,000	11.1	10.9	18.8	25,000
Cut-off Au Eq 3.0 g/t	**Tonnes**	**AuEq (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	215,000	9.8	9.7	9.3	67,000
Indicated	1,103,000	7.7	7.6	13.7	269,000
Measured + Indicated	1,318,000	8.1	7.9	13.0	336,000
Inferred	116,000	8.4	8.2	16.5	31,000
Cut-off Au Eq 1.0 g/t	**Tonnes**	**AuEq (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	340,000	6.9	6.8	6.8	74,000
Indicated	1,868,000	5.3	5.2	9.4	313,000
Measured + Indicated	2,208,000	5.6	5.5	9.0	388,000
Inferred	213,000	5.4	5.3	11.4	36,000
Cut-off Au Eq 0.75 g/t	**Tonnes**	**AuEq (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	359,000	6.6	6.5	6.5	75,000
Indicated	1,990,000	5.1	5.0	8.9	317,000
Measured + Indicated	2,349,000	5.3	5.2	8.6	392,000
Inferred	239,000	4.9	4.8	10.3	37,000
Cut-off Au Eq 0.50 g/t	**Tonnes**	**AuEq (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	380,000	6.2	6.2	6.2	75,000
Indicated	2,111,000	4.8	4.7	8.5	319,000
Measured + Indicated	2,491,000	5.0	4.9	8.1	394,000
Inferred	259,000	4.6	4.5	9.6	37,000

Table 14-11: Elk Gold Underground Resource Tabulation

Cut-off Au Eq 5.0 g/t	Tonnes	Au Eq (g/t)	Au Capped (g/t)	Ag Capped (g/t)	AuEq Ounces
Measured	5,000	7.48	7.42	5.70	1,000
Indicated	203,000	9.03	8.76	22.84	59,000
Measured + Indicated	208,000	8.99	8.73	22.43	60,000
Inferred	195,000	9.21	8.97	20.27	58,000
Cutoff AuEq 3.0 g/t	**Tonnes**	**AuEq (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	8,000	6.16	6.11	5.01	2,000
Indicated	406,000	6.45	6.25	17.52	84,000
Measured + Indicated	414,000	6.45	6.25	17.28	86,000
Inferred	304,000	7.29	7.08	17.06	71,000
Cutoff Au Eq 1.0 g/t	**Tonnes**	**Au Equivalent (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	11,000	4.74	4.69	4.04	2,000
Indicated	780,000	4.22	4.09	11.33	106,000
Measured + Indicated	792,000	4.23	4.10	11.22	108,000
Inferred	576,000	4.70	4.56	11.74	87,000
Cutoff Au Eq 0.75 g/t	**Tonnes**	**Au Equivalent (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	12,000	4.68	4.64	4.00	2,000
Indicated	870,000	3.87	3.75	10.40	108,000
Measured + Indicated	882,000	3.88	3.76	10.31	110,000
Inferred	647,000	4.28	4.15	10.79	89,000
Cutoff Au Eq 0.50 g/t	**Tonnes**	**Au Equivalent (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	14,000	3.92	3.88	3.42	2,000
Indicated	1,007,000	3.43	3.32	9.24	111,000
Measured + Indicated	1,022,000	3.44	3.33	9.16	113,000
Inferred	763,000	3.72	3.61	9.43	91,000

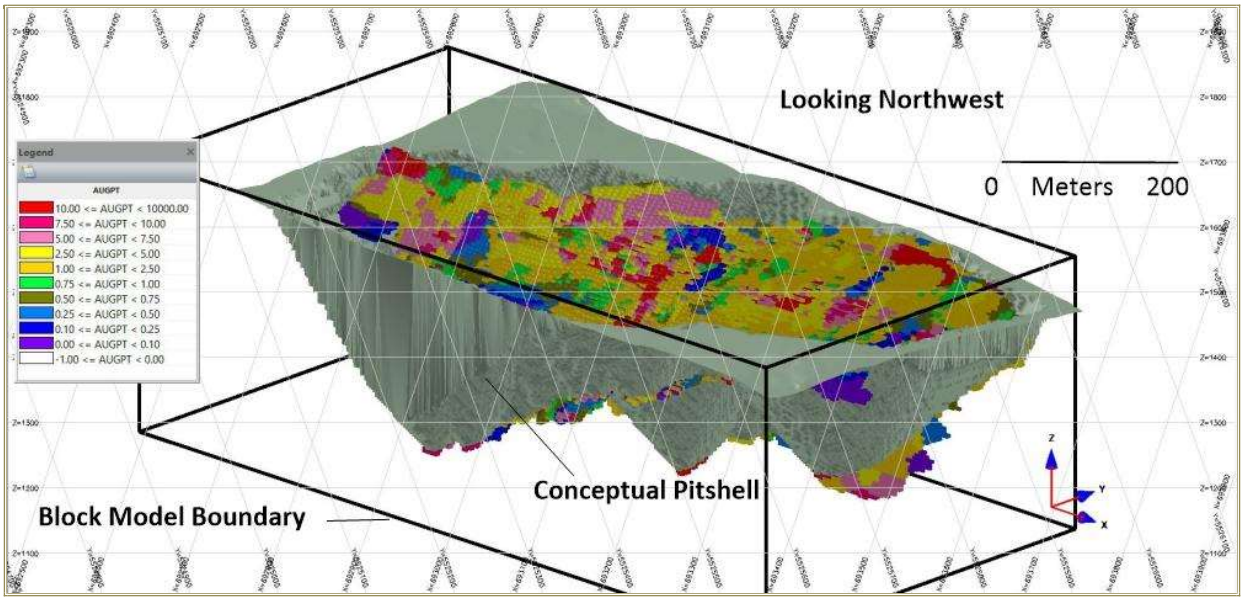
Notes: CIM definitions were followed for classification of Mineral Resources.
Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
Results are presented in situ and undiluted.
Mineral resources are reported at a cut-off grade of 0.5 g/t Au for pit-constrained resources and 5.0 g/t for underground resources.
The number of tonnes was rounded to the nearest thousand and the number of ounces of gold to the nearest hundred.
The Resource Estimate includes both gold and silver assays. The formula used to combine the metals is: AuEq = ((AU_Cap*41.8*0.96)+(Ag_Cap*0.55*0.86))/(41.8*0.96)
The resource estimate is effective as of July 3, 2020.

Figure 14-5 shows the block model in plan view. Figure 14-6 shows the block model in perspective view together with the conceptual pit shell. The view in Figure 14-6 is to the northwest. Representative cross sections are shown in Figure 14-7 to Figure 14-10. Figure 14-7 shows the location of the sections in plan view.



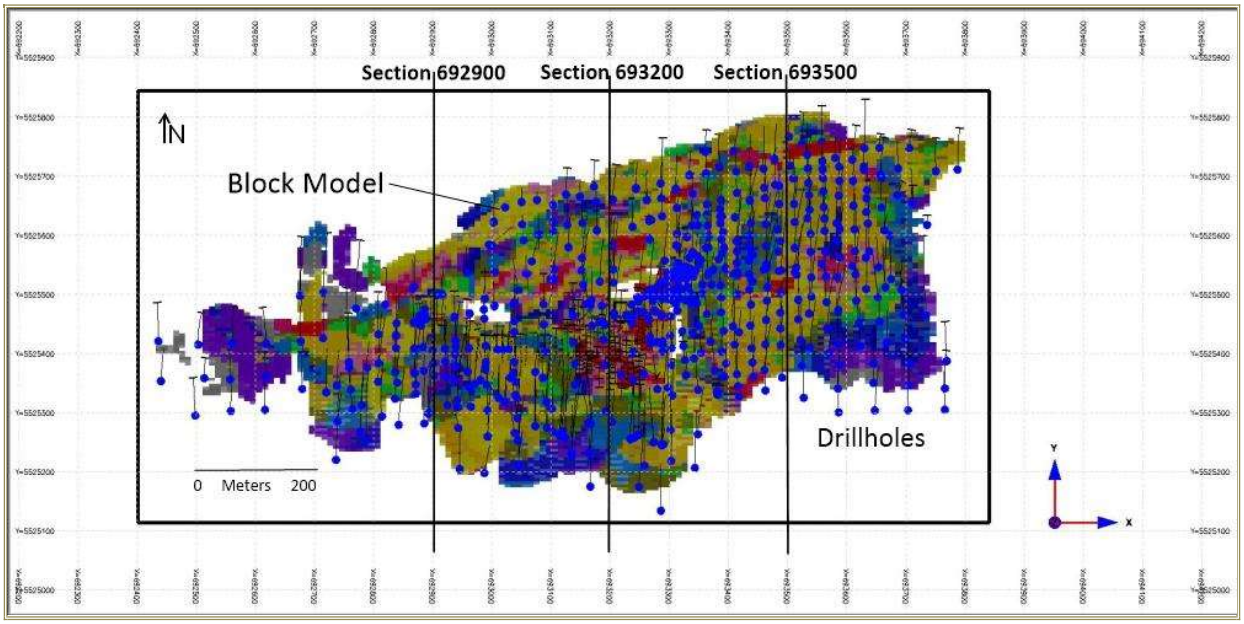
Source: GMRS, August 2020.

Figure 14-5: Elk Gold Block Model Plan View



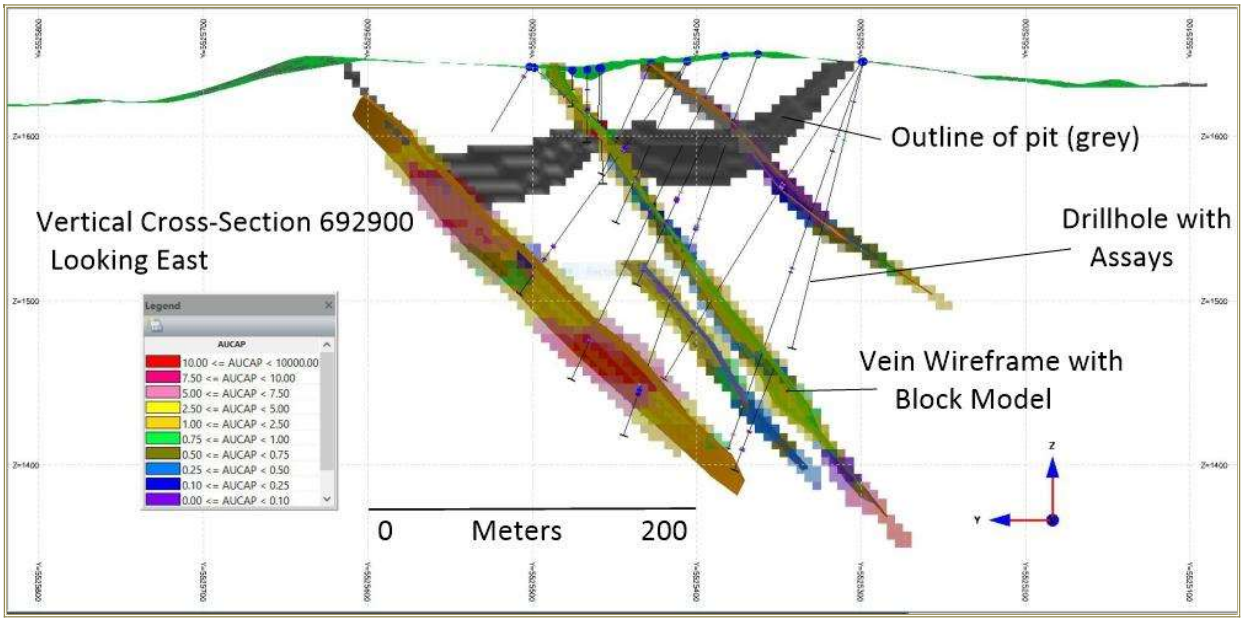
Source: GMRS, August 2020.

Figure 14-6: Elk Gold Block Model with Conceptual Pit Shell Perspective View



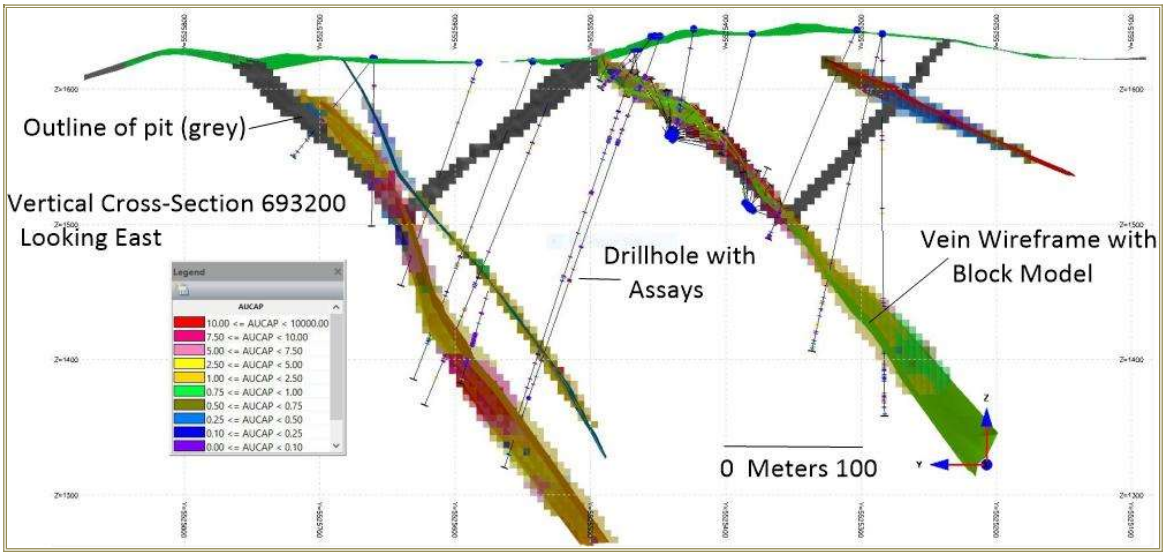
Source: GMRS, August 2020.

Figure 14-7: Cross Section Location in Plan View



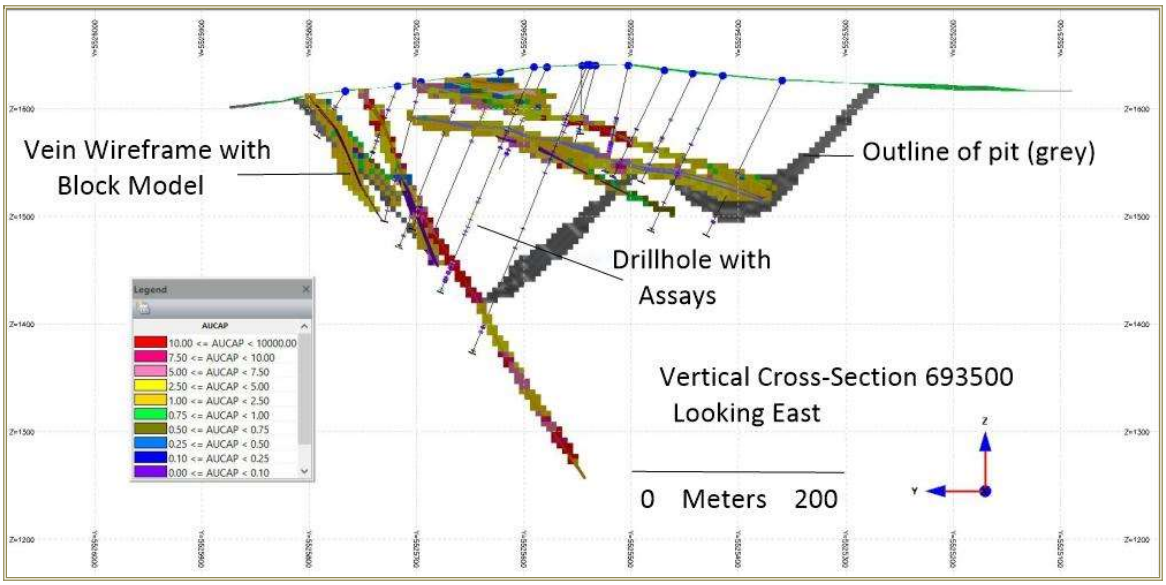
Source: GMRS, August 2020.

Figure 14-8: Cross Section 692900



Source: GMRS, August 2020.

Figure 14-9: Cross Section 693200



Source: GMRS, August 2020.

Figure 14-10: Cross Section 693500

14.13 Block Model Validation

The block model was validated in several ways: 1) visual inspection to ensure that block grades were consistent with nearby composite and assay grades, and 2) by use of swath plots. Swath plots are shown in Figure 14-11 for gold in Zone 1300, the largest of the zones and the one with the most data. The plots

show that the block model grades mimic the underlying composite grades but are less variable, as is to be expected because of the averaging of composite grades during the estimation process.

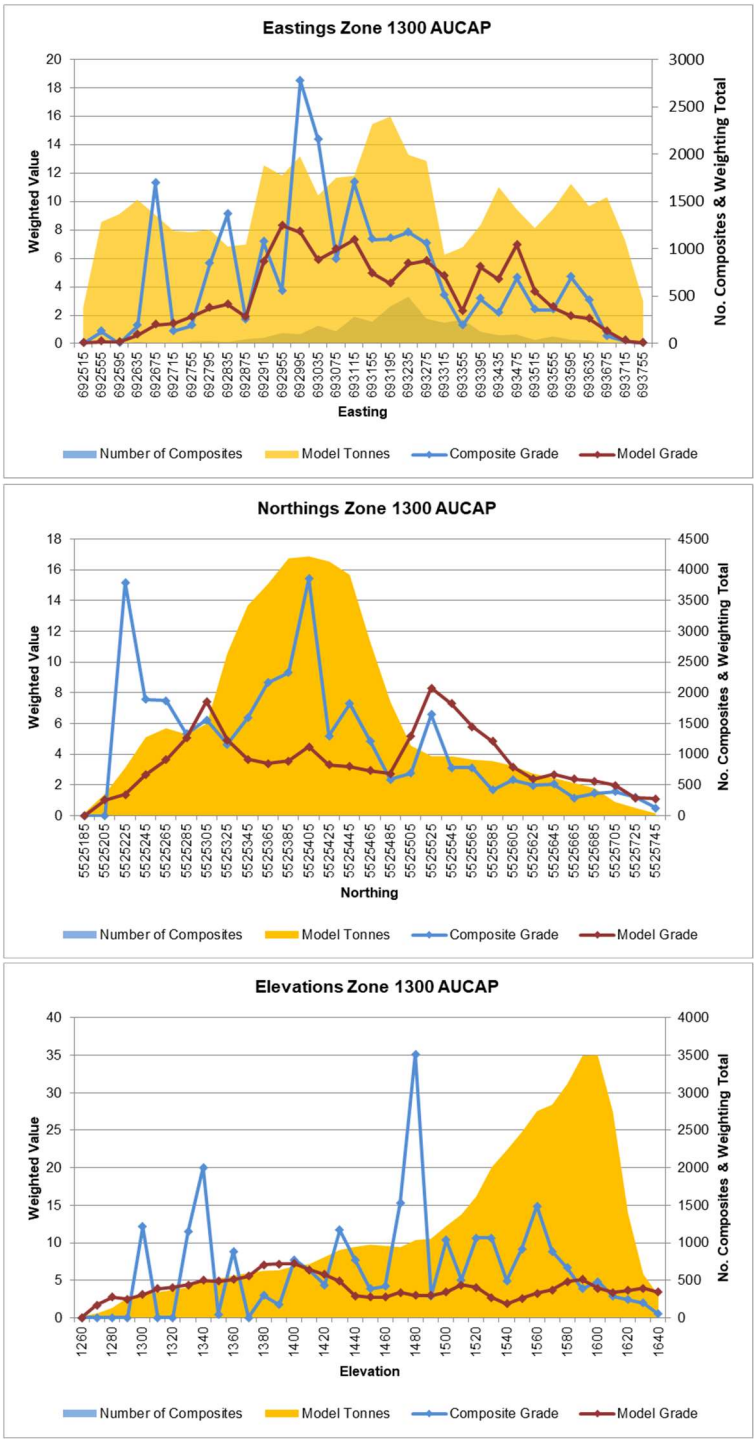


Figure 14-11: Elk Gold Swath Plot Capped Gold

14.14 Risk Factors

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the mineral resource estimate.

15 MINERAL RESERVES

Not applicable for a geologic resource.

16 MINING METHODS

16.1 Mine Plan Overview

The mine plan developed for the PEA was developed to expand upon the mine plan which was submitted for a mining permit amendment in May 2020. The Permit Mine Plan consisted of an open pit primary designed along the 1300 vein and mined at a rate of 200 t/d. The mineralized material was hauled off site for mineral processing, and as such no process plant or tailings facility was designed for the site.

The PEA mine plan presented herein continues the Permit Mine Plan for the first three years of operation and then proposes to construct a 900 t/d processing plant on site to reduce operating costs, increase gold production and to include more of the modelled resource. The expansion from 200 t/d to 900 t/d will require an Environmental Assessment certificate and an amendment to the M-199 mine permit after operation begins with the 200 t/d mine plan.

16.2 Open Pit Optimization

The limits of the ultimate pit were determined using the Lerch–Grossman (LG) pit optimization algorithm. The LG algorithm calculates the revenue and costs associated with each block in the block model using a series of economic, geotechnical, and unit cost input parameters.

The output of the algorithm is a series of nested pit shells which represent the break-even pit size over a range of gold prices. A break-even pit shell simply means that the pit has gone as deep as it can before the costs associated with going deeper outweigh the potential revenue gained from that material.

The input parameters used in the LG algorithm and the process of selecting an ultimate pit shell are described in the following subsections.

16.2.1 Input Parameters

The 3-D mineral resource block model, as developed by Global Mineral Resources Services, Greg Mosher, P.Geo., was used as the basis for deriving the economic shell limits for the Elk Gold Project. The block model dimensions were 10 m x 5 m x 5 m (X,Y,Z).

Estimates were made for gold prices, mining dilution, process recovery, offsite costs, and royalties. Mining, processing, and general administration operating costs were also estimated based on assumed processing throughput and, along with geotechnical parameters, formed the basis for open pit optimization (Table 16-1). The open pit mining costs assumed owner-operated mining.

Table 16-1: Mine Planning Optimization Input Parameters

Item	Units	Value
Au Price	US$/oz	1,361
Au Recovery	%	92
Au Selling Cost	%	2
NSR Royalty	%	2
Mining Cost	US$/t	2
Processing and G&A Cost	US$/t milled	32
Pit Slope Angles	overall degrees	45
Mining Dilution	%	Internal
Mining Recovery	%	100
Processing Rate	t/d	900

The mining dilution is assumed to be entirely internal dilution included in the resource modelling methodology. This is supported by the relative increase in actual run-of-mine grades realized during the 2012–2014 bulk sampling program compared to the resource model grades. The resource model is constrained to a wireframe constructed from 0.5 g/t drill-hole composite grades, and as such includes material across the boundaries of the actual mineralized quartz veins. The bulk sample program, as described in Section 24, was able to mine more selectively than the resource model envisioned and achieved higher grades than predicted in the block model. The proposed Elk Gold mining operation is anticipated to be less selective than the bulk mining program to be more productive; however, it is still selective enough to achieve the modelled resource grades without additional external dilution.

The mineral inventory block model for the Elk Gold deposit was then transferred to an open pit optimization software to determine optimal mining shells. This evaluation included the aforementioned parameters.

The economic shell limits included Measured, Indicated, and Inferred mineral resources. Inferred mineral resources are considered too speculative, geologically, to have the economic considerations applied to them to be categorized as Mineral Reserves. There is no certainty that the Inferred mineral resources will be upgraded to a higher resource category.

16.2.2 Optimization Cut-off Grade

For the pit optimization process, which was undertaken with Whittle™ (4.7) mine optimization software, mill feed selection was determined using the pit optimization software. With this method, mineralized material is selected by comparing the cash flow (calculated from the block Au grades) which would be produced by processing it and the cash flow which would be produced by mining it as non-mineralized material. If the cash flow from processing the mineralized material is higher, the material is treated as mill feed. If not, it is treated as non-mineralized material. In this particular case

mill feed selection by cash flow will produce the same result as that produced by the use of marginal cut-offs.

16.2.3 Optimization Results

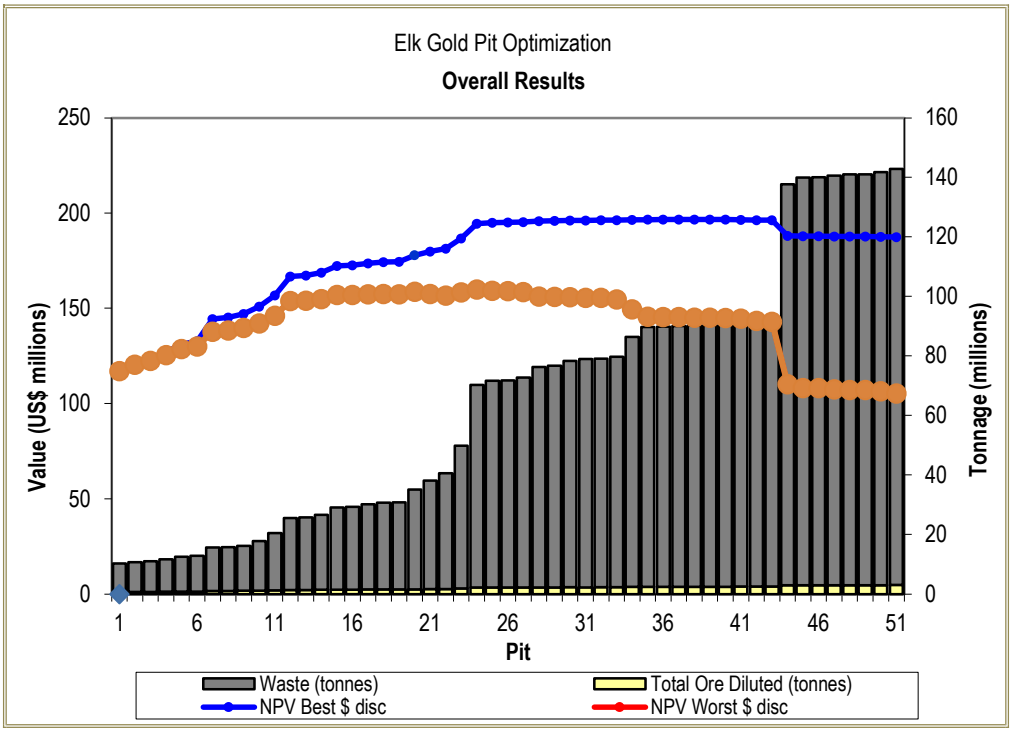
A series of optimized shells were generated for the Elk Gold deposit based on varying revenue factors. The results were analyzed with shells chosen as the basis for ultimate limits and preliminary phase selection.

The results of the pit optimization evaluation on the deposit for varying revenue factors values are summarized in Table 16-2 and Figure 16-1 to Figure 16-3 for Inferred, Indicated, and Measured mineral resources. Note the NPV in this optimization summary does not consider capital costs and is used only as a guide in shell selection and determination of the mining shapes. The actual cash flow NPV of the Project is summarized in Section 22 of this report.

Table 16-2: Overall Optimization Results

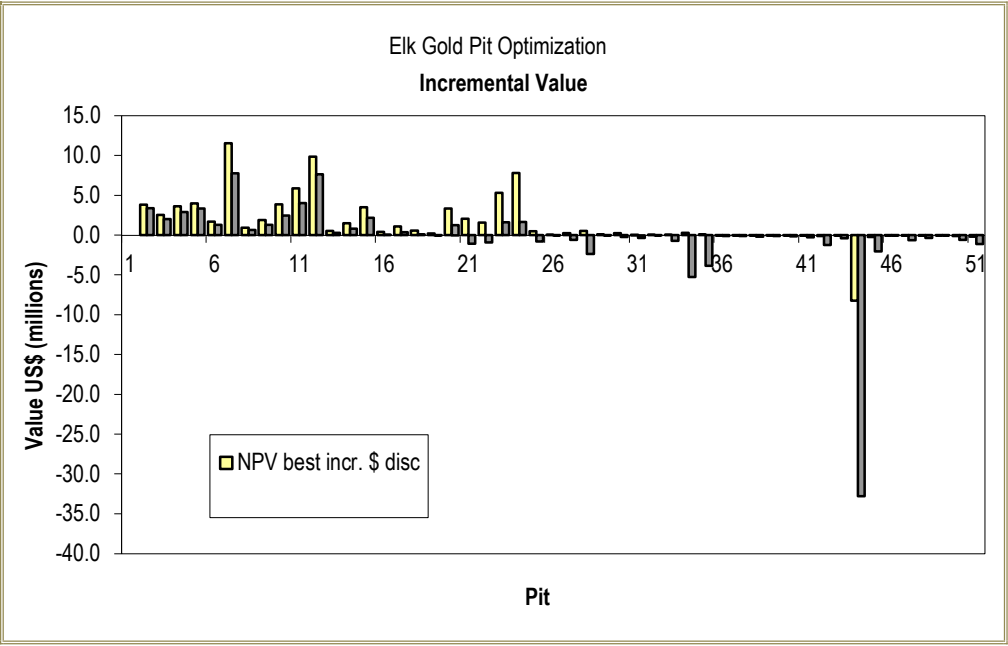
Final Pit	Revenue Factor	Mine Life	Total Ore Diluted			Waste	Strip Ratio	Total	NPV Best	NPV Worst
			(t)	Au (g/t)	Au g	(t)		(t)	$ Disc	$ Disc
1	0.30	3.3	625,046	7.15	4,469,748	9,584,101	15.3	10,209,147	117,171,012	117,171,012
2	0.32	3.5	670,527	6.94	4,651,698	9,965,357	14.8	10,635,884	120,977,600	120,533,025
3	0.34	3.6	699,421	6.83	4,776,893	10,302,103	14.7	11,001,524	123,505,914	122,557,163
4	0.36	3.8	739,318	6.71	4,963,726	10,923,889	14.7	11,663,207	127,127,656	125,459,865
5	0.38	4.0	784,907	6.60	5,176,974	11,703,048	14.9	12,487,955	131,089,742	128,793,128
6	0.40	4.1	807,854	6.52	5,266,314	12,012,814	14.8	12,820,668	132,764,987	130,070,177
7	0.42	5.0	992,963	5.99	5,946,734	14,585,988	14.6	15,578,951	144,309,974	137,813,685
8	0.44	5.1	1,013,031	5.93	6,002,364	14,736,000	14.5	15,749,031	145,218,087	138,465,346
9	0.46	5.3	1,052,992	5.81	6,122,287	15,140,153	14.3	16,193,145	147,111,448	139,762,992
10	0.48	5.5	1,103,355	5.78	6,375,582	16,630,839	15.0	17,734,194	150,963,027	142,203,302
11	0.50	5.9	1,170,437	5.79	6,778,953	19,301,330	16.4	20,471,767	156,840,656	146,227,981
12	0.52	6.7	1,327,284	5.68	7,535,768	24,173,656	18.2	25,500,940	166,679,492	153,868,157
13	0.54	6.8	1,341,968	5.65	7,582,144	24,410,332	18.1	25,752,300	167,220,993	154,153,159
14	0.56	7.0	1,383,167	5.58	7,716,506	25,159,787	18.1	26,542,954	168,702,177	154,952,028
15	0.58	7.3	1,443,056	5.56	8,028,149	27,651,872	19.1	29,094,928	172,193,991	157,127,807
16	0.60	7.3	1,459,174	5.53	8,065,913	27,811,826	19.0	29,271,000	172,594,451	157,153,868
17	0.62	7.5	1,483,392	5.51	8,171,648	28,646,619	19.3	30,130,011	173,666,539	157,498,652
18	0.64	7.5	1,498,681	5.49	8,232,664	29,135,961	19.4	30,634,642	174,241,254	157,563,996
19	0.66	7.6	1,506,347	5.48	8,254,450	29,281,668	19.4	30,788,015	174,430,648	157,522,180
20	0.68	8.2	1,566,015	5.53	8,662,181	33,556,532	21.4	35,122,547	177,748,559	158,804,236
21	0.70	8.6	1,619,827	5.52	8,948,399	36,497,463	22.5	38,117,290	179,819,652	157,701,820
22	0.72	8.9	1,668,790	5.50	9,186,523	38,926,746	23.3	40,595,536	181,403,957	156,786,340
23	0.74	10.3	1,840,385	5.46	10,041,596	47,998,904	26.0	49,839,289	186,706,690	158,405,218
24	0.76	13.3	2,122,137	5.55	11,771,856	68,089,123	32.0	70,211,260	194,498,928	160,061,503

Final Pit	Revenue Factor	Mine Life	Total Ore Diluted			Waste (t)	Strip Ratio	Total (t)	NPV Best $ Disc	NPV Worst $ Disc
			(t)	Au (g/t)	Au g					
25	0.78	13.4	2,144,237	5.55	11,892,241	69,491,901	32.4	71,636,138	194,986,575	159,278,390
26	0.80	13.4	2,149,171	5.54	11,906,184	69,619,855	32.3	71,769,026	195,029,225	159,220,295
27	0.82	13.5	2,167,011	5.53	11,986,785	70,580,041	32.5	72,747,052	195,271,112	158,621,775
28	0.84	14.0	2,197,816	5.57	12,236,798	74,040,752	33.6	76,238,568	195,775,122	156,236,183
29	0.86	14.1	2,204,908	5.57	12,273,657	74,528,641	33.8	76,733,549	195,870,726	156,181,155
30	0.88	14.3	2,223,530	5.57	12,385,890	76,091,311	34.2	78,314,841	196,119,685	155,941,802
31	0.90	14.4	2,232,775	5.57	12,432,615	76,741,352	34.3	78,974,127	196,196,841	155,608,804
32	0.92	14.4	2,235,114	5.57	12,439,571	76,823,227	34.3	79,058,341	196,207,214	155,550,064
33	0.94	14.5	2,242,761	5.57	12,483,541	77,482,143	34.5	79,724,904	196,249,006	154,843,986
34	**0.96**	**15.5**	**2,359,194**	**5.49**	**12,945,321**	**83,983,763**	**35.6**	**86,342,957**	**196,515,751**	**149,566,061**
35	0.98	15.9	2,415,793	5.45	13,171,978	87,253,303	36.1	89,669,096	196,582,828	145,715,560
36	**1.00**	**16.0**	**2,417,573**	**5.45**	**13,183,719**	**87,446,485**	**36.1**	**89,864,058**	**196,581,927**	**145,576,380**
37	1.02	16.0	2,420,038	5.45	13,193,149	87,585,752	36.1	90,005,790	196,577,240	145,471,243
38	1.04	16.0	2,424,405	5.45	13,208,798	87,822,241	36.2	90,246,646	196,568,778	145,261,921
39	1.06	16.0	2,426,602	5.45	13,214,646	87,902,321	36.2	90,328,923	196,563,548	145,157,051
40	1.08	16.1	2,428,802	5.45	13,225,136	88,081,199	36.2	90,510,001	196,551,126	145,014,789
41	1.10	16.1	2,439,418	5.44	13,260,641	88,653,225	36.3	91,092,643	196,490,663	144,717,095
42	1.12	16.3	2,453,936	5.43	13,326,541	89,828,272	36.6	92,282,208	196,357,153	143,492,443
43	1.14	16.4	2,462,218	5.42	13,346,232	90,117,860	36.6	92,580,078	196,312,633	143,083,031
44	1.16	23.0	2,919,151	5.38	15,717,471	134,799,989	46.1	137,719,140	188,071,226	110,295,544
45	1.18	23.1	2,940,370	5.38	15,832,069	137,021,885	46.6	139,962,255	187,813,405	108,261,304
46	1.20	23.1	2,941,356	5.38	15,836,534	137,106,652	46.6	140,048,008	187,802,260	108,207,361
47	1.22	23.2	2,951,904	5.38	15,867,789	137,650,239	46.6	140,602,143	187,721,278	107,558,343
48	1.24	23.3	2,956,669	5.37	15,888,251	138,059,809	46.6	141,016,478	187,659,171	107,188,510
49	1.26	23.3	2,957,934	5.37	15,890,547	138,094,486	46.6	141,052,420	187,651,512	107,157,747
50	1.28	23.4	2,964,534	5.37	15,921,687	138,748,490	46.8	141,713,024	187,545,017	106,544,805
51	1.30	23.5	2,978,534	5.36	15,973,880	139,805,630	46.9	142,784,164	187,360,518	105,437,272



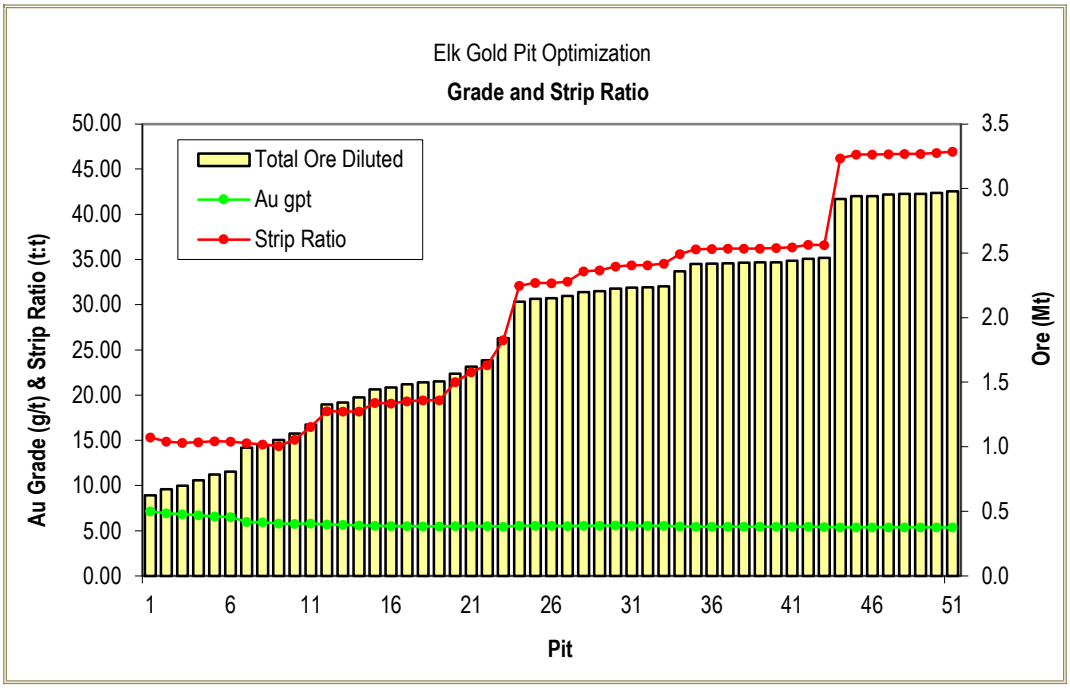
Source: AKF Mining Services, August 2020.

Figure 16-1: Open Pit Optimization Cumulative Results



Source: AKF Mining Services, August 2020.

Figure 16-2: Open Pit Optimization Incremental Value Results



Source: AKF Mining Services, August 2020.

Figure 16-3: Open Pit Optimization Incremental Tonnage Results

The pit optimization software produces two initial scheduling scenarios: one in which each nested pit shell is mined sequentially and one where the pit is mined bench-by-bench all the way to the ultimate pit limit. The first scenario releases the highest-value material faster and delays non-mineralized material stripping costs until later in the mine plan; however, the nested pit shells are often too narrow to actually mine. The second scenario advances the non-mineralized material stripping costs and delays the release of the highest-value materials, and as such, presents a more pessimistic project value. The true project can be assumed to be somewhere between those two book-ends.

For the Elk Gold deposit, Pit shell #34 at revenue factor 0.96 represents the last incremental increase in value for the "Best Case" project value and it also contains over 400,000 oz of gold, which represents a scale of mining operation that demonstrates the potential value of the Elk Gold Project.

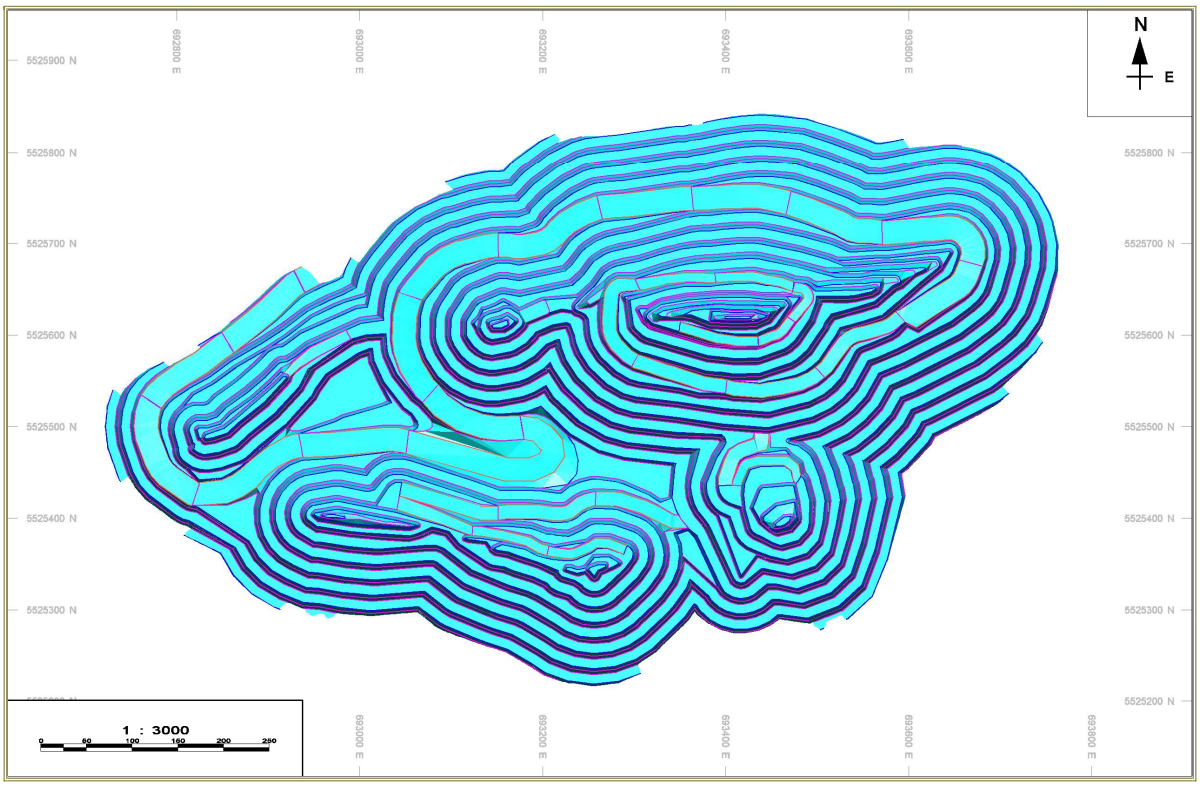
Pit shell #34 includes almost 2.4 Mt of mineralized material above cut-off, which represents an 11-year mine life when accounting for the three-year toll milling period. Of the plant feed material, 14% is classified as Measured, 77% is classified as Indicated, and 9% is classified as Inferred. Note that Inferred material is consider too geologically speculative to have economic value applied to it, and there is no guarantee that Inferred resources will convert to Indicated or Measured.

16.3 Open Pit Mine Design

For the Elk deposit, previous mine design and mine planning work for the Permit Mine Plan resulted in a series of phased pit designs which were employed again in the early phases of the PEA Mine Plan. Those pit designs were combined with nested pit shells in the mine planning software to create a series of pit phases which exploit high value and low strip ratio mineralized material early in the mine plan and also serve to delay non-mineralized material stripping costs until later in the mine life.

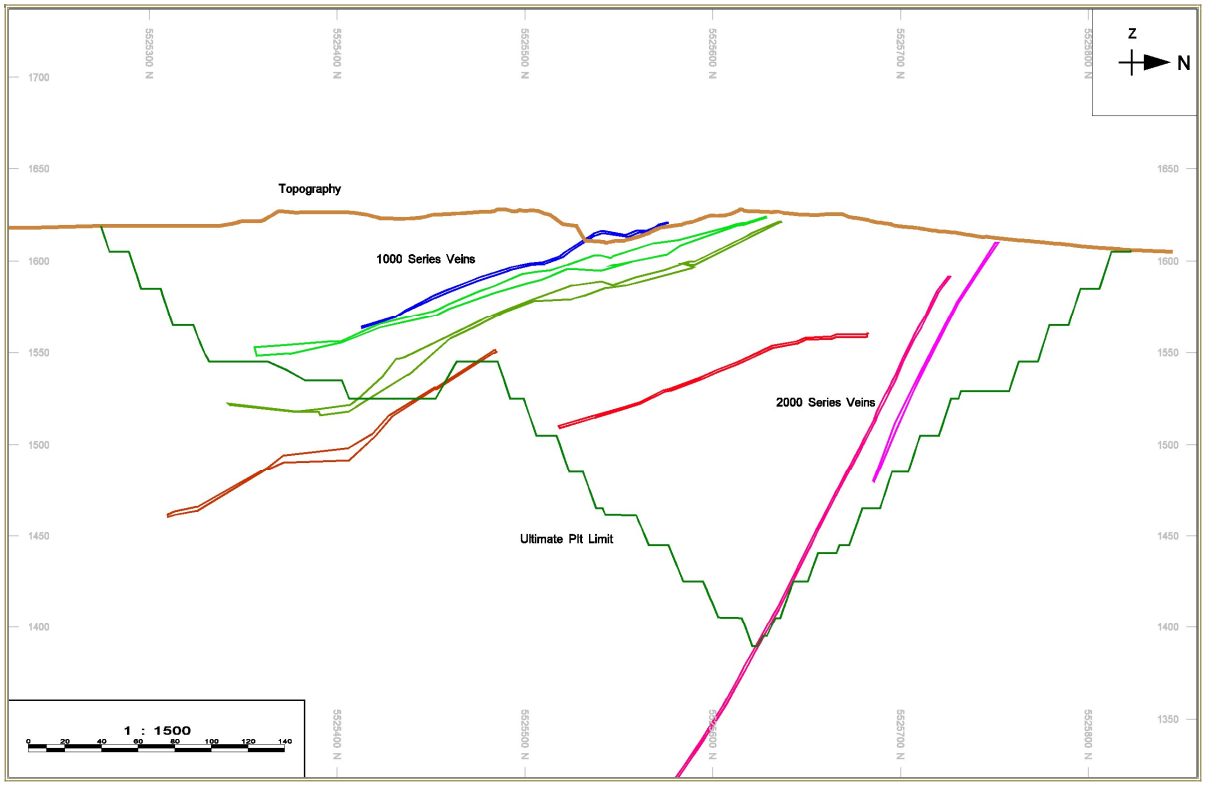
A detailed ultimate pit design was developed to demonstrate the achievability of the final whittle shell. The pit design achieved a 1% increase in mineralized material above cut-off, a 3% increase in non-mineralized material mined and 2.5% decrease in contained gold ounces. This variance from the whittle shell volumes was considered acceptable. The production schedule however is based on the volumes of the Whittle shell.

Figure 16-4 and Figure 16-5 represent plan and section views of the ultimate pit shape.



Source: AKF Mining Services, August 2020.

Figure 16-4: Plan View of Elk Gold Ultimate Pit Design



Source: AKF Mining Services, August 2020.

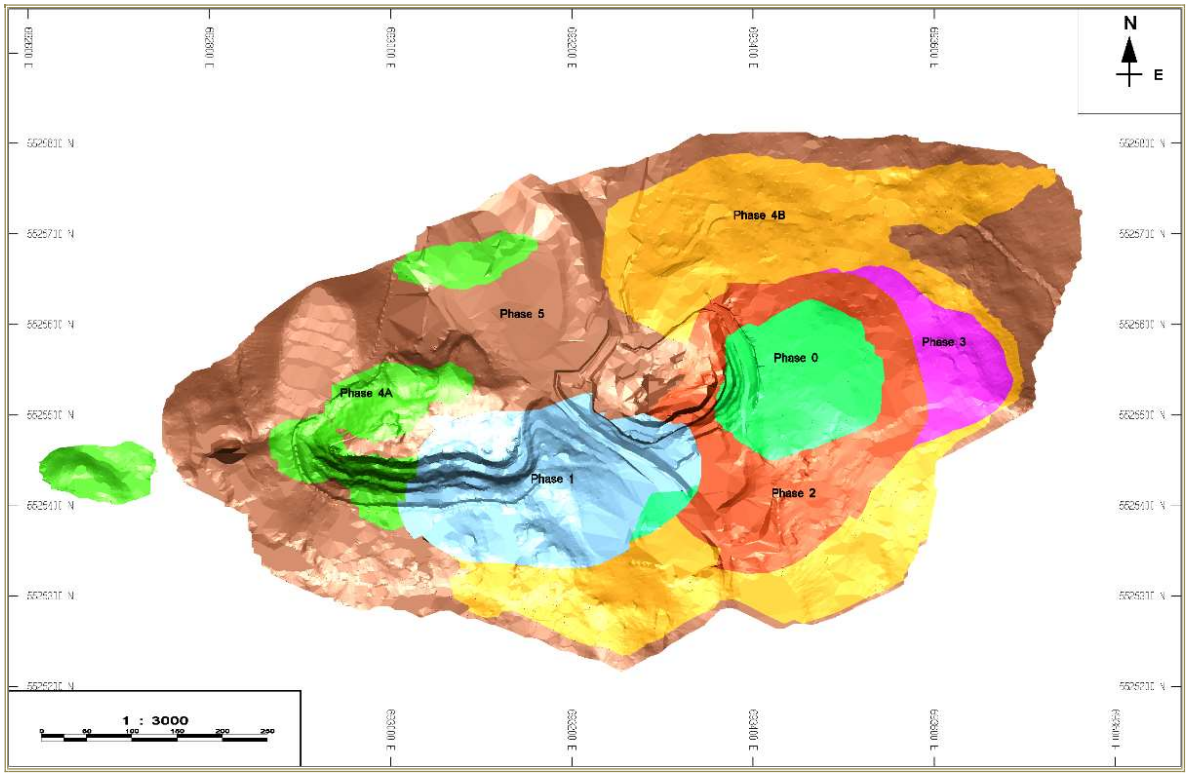
Figure 16-5: Typical Long Section Through the Ultimate Pit Design

16.4 Mine Sequence/Phasing

The pit phasing and mining sequence for the Elk Gold Project is determined by targeting the highest grade and lowest strip-ratio areas of the deposit to be mined first, along with the consideration that the first three years of mining are constrained to the footprint of the Permit Mine Plan. The Permit Mine Plan is designed along the 1300 vein and includes a series of detailed pit phase designs. These designs were employed for the PEA Mine Plan along with nested Whittle shells.

The pit optimization exercise indicated that there was a section of the 1300 vein which sat with an existing Permit Mine Plan phase design that was higher grade and lower strip-ratio than any of the other phases, so this Whittle shell was selected to be the starter pit.

After the starter pit, the Permit Mine Plan designs for Phases 1, 2, and 3 were mined. And following those designs the next phase was the revenue-factor 0.68 Pit shell #20, before the ultimate Pit shell, #34, was eventually mined.



Source: AKF Mining Services, August 2020.

Figure 16-6: Elk Gold Pit Phases for Scheduling

16.5 Rock Management Storage Facility Design

Material below the cut-off grade is stored in one rock management storage facility (RMSF) located to the west of the open pit. The RMSF was designed with sufficient capacity to store the 84 Mt of non-mineralized material, assuming a loose density of 1.8 t/m^3.

The WSF will be constructed in a series of 20 m lifts with 37° face slopes and catch berms on each lift, resulting in an overall slope of 26°.

16.6 Open Pit Mine Operation

The open pit mining activities for the Elk Gold pit were assumed to be primarily undertaken by a rental-leased owner-operated fleet as the basis for this preliminary economic assessment. The average unit mining costs used in the Project economics was $3.03/t of material mined, for pit and dump operations, road maintenance, mine supervision, and technical services. The cost estimate was built from first principles and based on experience of similar sized open pit operations and local conditions. The open pit mining costs consider variations in haulage profiles and equipment selection.

Labour rates were estimated using local information.

16.6.1 Equipment

Table 16-3 summarizes the assumed, all diesel, major open pit equipment requirements used for the basis of this study and are based on similar-sized open pit operations. The proposed processing rate of 324,000 t/a was used, along with deposit and pit geometry constraints, to estimate the mining equipment fleet needed. The fleet has an estimated maximum capacity of 39,000 t/d total material, which would be sufficient for the life-of-mine (LOM) plan.

Table 16-3: Major Open pit Equipment Requirements

Equipment Type	# of Units
250 mm dia. Rotary, Crawler Drill	2
5 m^3 Front Shovel	1
10 m^3 Front Shovel	2
136-ton Haul Truck	9
D10-class 17.3 ft blade	2
834H-class 15.2 ft blade	1
16H-class Grader	1
90-ton Water Truck	1

16.6.1.1 Unit Operations

The 250 mm diameter blast hole drills are planned to perform all of the production drilling in the mine. The drills are anticipated to drill up to the mineralized zones to precondition them for mining; however, the mineralized zones themselves will not be drilled and blasted to reduce dilution and ore loss. The material is anticipated to be sufficiently fractured as to allow the excavator to dig it without additional blasting.

The main non-mineralized material loading and haulage fleet is planned to consist of 136 tonne haul trucks, loaded primarily with the diesel-powered 10 m^3 front shovels. The mineralized material will be mined by the smaller 5 m^3 shovel which will allow for more selectivity.

As pit conditions dictate, the fleet of ancillary equipment, including track dozers, a grader, rubber-tire dozer, and water truck are planned to clean and maintain active mining benches, haul roads, and develop the non-mineralized rock storage areas.

The following additional equipment will be required to support mining operations:

- Explosives storage and delivery equipment
- Field maintenance vehicles
- Light vehicles for personnel transportation
- Light plants
- Portable aggregate plant used for plant feed sampling and assaying
- Utility excavator.

16.7 Mine Schedule

The maximum planned total material mining-rate during the LOM is approximately 47,000 t/d, while the average total mining rate was planned to be 23,000 t/d.

Measured, Indicated, and Inferred mineral resources were used in the LOM plan, with Measured and Indicated mineral resources making up 91% of the total LOM tonnage processed. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves and there is no certainty that the preliminary economic assessment will be realized. There is no certainty that the Inferred mineral resources will be upgraded to a higher resource category.

Table 16-4 is a summary of total material movement by year for the LOM production schedule.

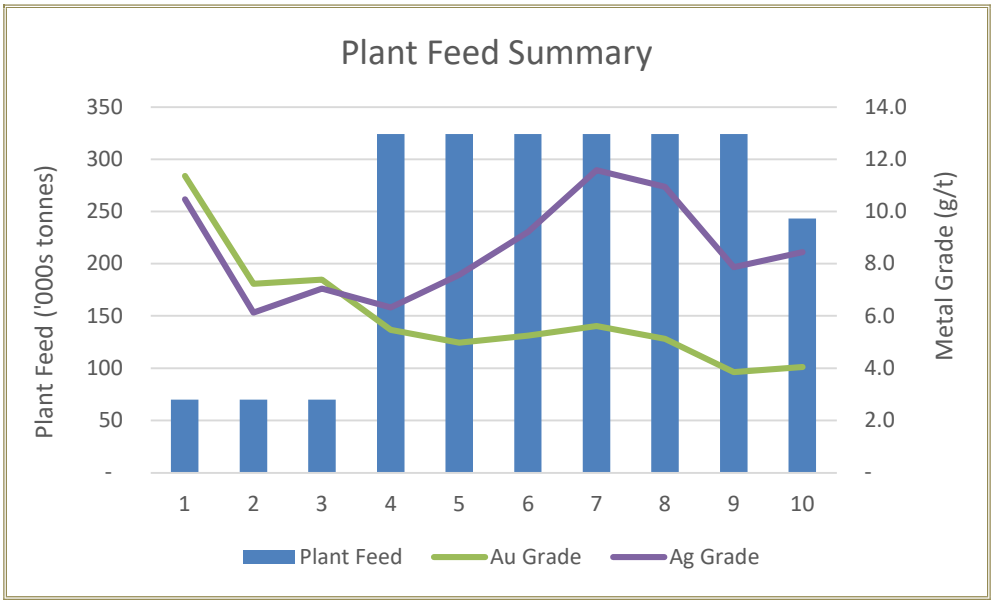
Table 16-4: Proposed LOM Open Pit Production Schedule

		Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Calendar Days	#	**3,652**	365	365	365	366	365	365	365	366	365	365
Plant Feed	dmt ('000s)	**2,397**	70	70	70	324	324	324	324	324	324	243
Non-Plant Feed	dmt ('000s)	**83,617**	2,056	1,940	2,358	3,003	4,041	13,749	14,126	17,075	16,492	8,777
Au	(g/t)	**5.26**	11.36	7.23	7.38	5.47	4.97	5.24	5.61	5.11	3.85	4.04
Ag	(g/t)	**8.77**	10.45	6.13	7.04	6.32	7.57	9.21	11.58	10.93	7.86	8.43
S	(%)	**0.71**	0.81	0.08	0.10	0.70	0.45	0.70	0.99	0.78	0.83	0.82
Strip Ratio	(np:p)	**34.9**	29.4	27.7	33.7	9.3	12.5	42.4	43.6	52.7	50.9	36.1
Au Mined	oz ('000s)	**406**	25.6	16.3	16.6	57.0	51.9	54.7	58.5	53.3	40.1	31.6
Au Recovered	oz ('000s)	**373**	23.5	15.0	15.3	52.4	47.7	50.3	53.8	49.1	36.9	29.1
Ag Mined	oz ('000s)	**677**	23.5	13.8	15.9	65.9	78.9	96.0	120.7	113.9	81.9	66.0
Ag Recovered	oz ('000s)	**609**	21.2	12.4	14.3	59.3	71.0	86.4	108.6	102.5	73.7	59.4

The Elk Gold deposit is planned to produce a total of 2.4 Mt of plant process feed material and 83 Mt of non-plant feed (34.9:1 overall strip ratio) over a 10-year mine operating life. The current LOM plan focuses on mining higher-grade material early in the schedule and balancing grade and strip ratios, while trying to maximize NPV. A low-grade stockpile is developed to maximize the grade of material processed early in the mine life. The low-grade material is processed at the end of the mine life.

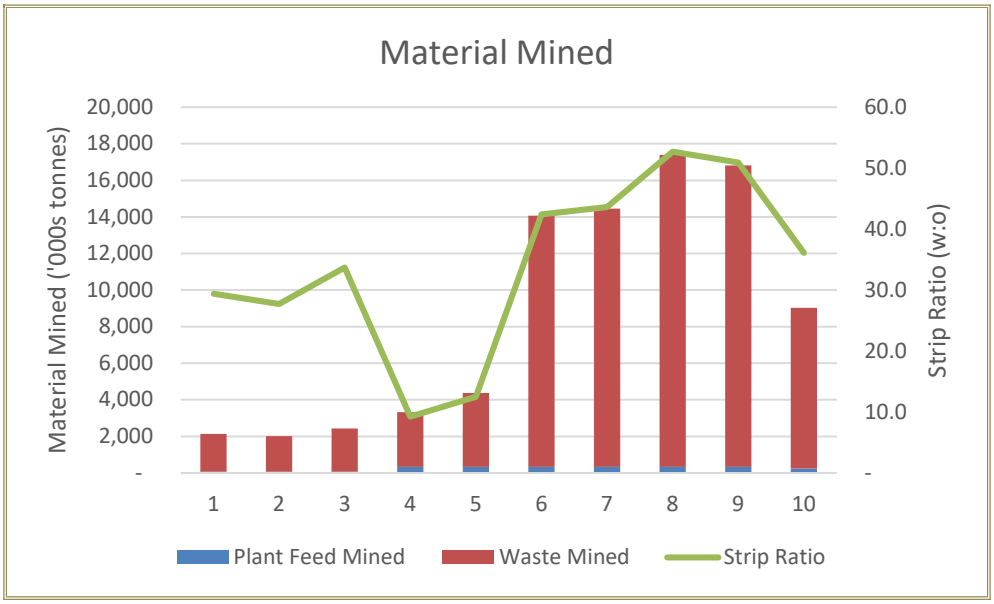
Figure 16-7summarizes process tonnages and metal grades by period. Figure 16-8 illustrates the feed tonnage by phase and period, as well as overall gold grades. The mine is estimated to produce an average of 50,000 oz of gold annually for Year 4 through Year 10.

Figure 16-9 represents the proposed overall site layout for the Elk Gold Project.



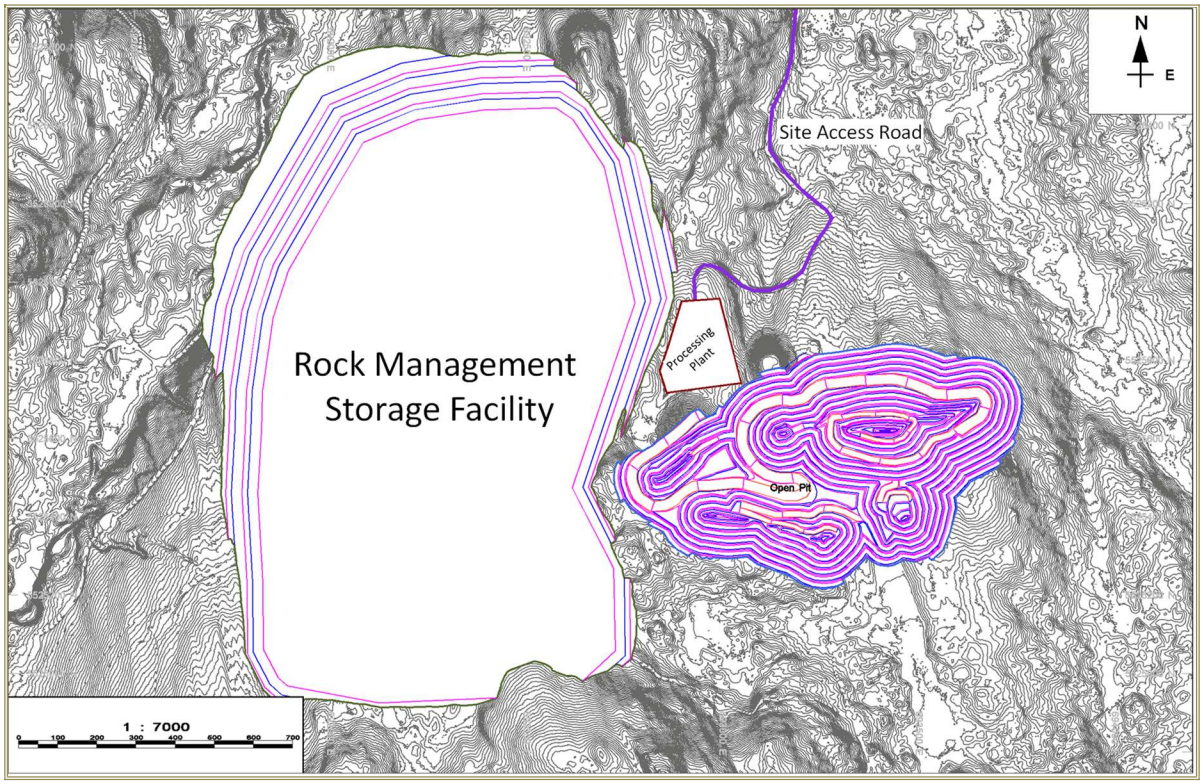
Source: AKF Mining Services, August 2020

Figure 16-7: Plant Feed and Metal Grades by Period



Source: AKF Mining Services, August 2020

Figure 16-8: Material Mined and Strip Ratio by Period



Source: AKF Mining Services, August 2020

Figure 16-9: Overall Site Plan

The total plant feed reported by resource category is summarized in Table 16-5.

Table 16-5: Minable Resources by Category

Category	Tonnes ('000s)	Au (g/t)	Ag (g/t)
Measured	199	5.1	11.1
Indicated	1,849	5.1	9.0
Inferred	348	6.4	6.4
Total	**2,397**	**5.3**	**8.8**

16.7.1.1 Mine Development Schedule

Year 1—The first year of mining includes two months of site preparation. The preproduction activities include developing the new water settling pond below the RMSF, and associated collection ditches. It also includes stripping organic material, topsoil, and till from the initial footprint of the RMSF and open pit Phase 0. It also includes mobilizing the initial fleet of mobile equipment, modular office facilities, and explosives storage.

The remainder of Year 1 will be the initial year of mining production, including 70,000 tonnes of mineralized material shipped to the toll milling facility.

Years 2 and 3—The mine will continue to mine the initial phases of the open pit and ship 70,000 t/a of mineralized material to the toll milling facility. The mine will also initiate an Environmental Assessment process which is required to expand mine production in subsequent years, as well as apply for a mine permit amendment for the expanded mining rate. In Year 3 it is anticipated that construction will take place to build a 900 t/d mill on site.

Years 4 to 10—The mining rate will increase to 324,000 t/a of plant feed for the 900 t/d mill on site. Tailings material will be dewatered and comingled with non-mineralized rock in the RMSF, removing the requirement to construct a tailings storage facility. The mining rate will gradually increase over the mine life as the required strip ratio also increases.

Year 11—The final year of production mining will also include the initiation of major reclamation activities. The mine will prepare ahead of time for the ultimate reclamation and closure of the facility.

Year 12—Major mine site reclamation activities will be completed.

16.8 Mining Geotechnical Information

The pit optimization was carried out using 45° overall slope angles which include allowances for haul roads.

The detailed pit design for the ultimate pit was designed using 50° inter-ramp slope angles and 20 m benches. The overall slope angles achieved in the detailed design are generally in line with the 45° pit shell slopes.

Geotechnical characterization completed on the Property to date has been focused on the shallower pits designed for the Permit Mine Plan. Additional geotechnical drilling, kinematic assessment, and stability analysis is recommended for considering the larger open pit designed for this PEA.

16.9 Surface Water Management

Surface water management will include the control of waters that come into contact with mine facilities, the diversion of non-contact water to avoid mine facilities and maintaining the supply of water required for mineral processing.

The Elk Gold Project is located in the upper reaches of the Siwash Creek watershed, and more specifically it is located in the watersheds of the short tributaries Bullion Creek and Don Creek. As the Project is at the top of the watershed, there is limited application for using diversion ditches to keep fresh water from contacting mine facilities. The WMSF is the only facility where diversion ditches will be deployed.

The open pit, processing plant, and RMSF will all have collection ditches to control surface water runoff and direct it into settling ponds for polishing before discharge into the receiving environment. The settling ponds will also receive water from pit dewatering operations.

The mineral processing plant will maintain a water supply and maximize the recycling of process water to minimize the volume of affected water. Further analysis is required to model the quality of the process water and any mitigations that may be required prior to discharging that water into the receiving environment.

No active water treatment is envisioned for the site. The water discharged from the RMSF is anticipated to have nitrogen levels potentially higher that water quality guidelines, which will be mitigated via a constructed wetland in the lower sections of the Don Creek channel.

17 RECOVERY METHODS

17.1 Introduction

The Process Design Criteria and Mass Balance detail the annual production capabilities, major mass flows and capacities, and availability for the process plant. Freeform will be toll-treating the mineralized material from the mine for the first few years of operation, with the intent of building a flotation plant at site for the start of Year 4. The consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in Section 21, as outlined in the toll treatment agreement. Key process design criteria from Section 13 and design considerations for a gravity flotation circuit are summarized in Table 17-1. The following sections describe the flowsheet design and equipment parameters required to achieve a saleable gold concentrate.

The process plant will include:

- Two stages of crushing
- A single ball mill, grinding in reverse closed-circuit, with cyclones and a gravity circuit
- Rougher flotation and one stage of cleaner flotation
- Concentrate dewatering using thickeners and pressure filters
- Concentrate storage and load-out facilities.

The crushing plant will have a throughput of 900 t/d, with an average LOM head grade of 5.26 g/t Au and 8.77 g/t Ag. The crushing circuit will operate for 12 hours per day, with an hourly throughput of 75 t/h. The grinding and flotation circuits will operate 24 hours per day, 365 days per year, with an availability of 90%, and a design throughput of 42 t/h.

17.2 Plant Design Criteria

The basis for design and major equipment sizing is listed below in Table 17-1.

Table 17-1: Process Design Criteria

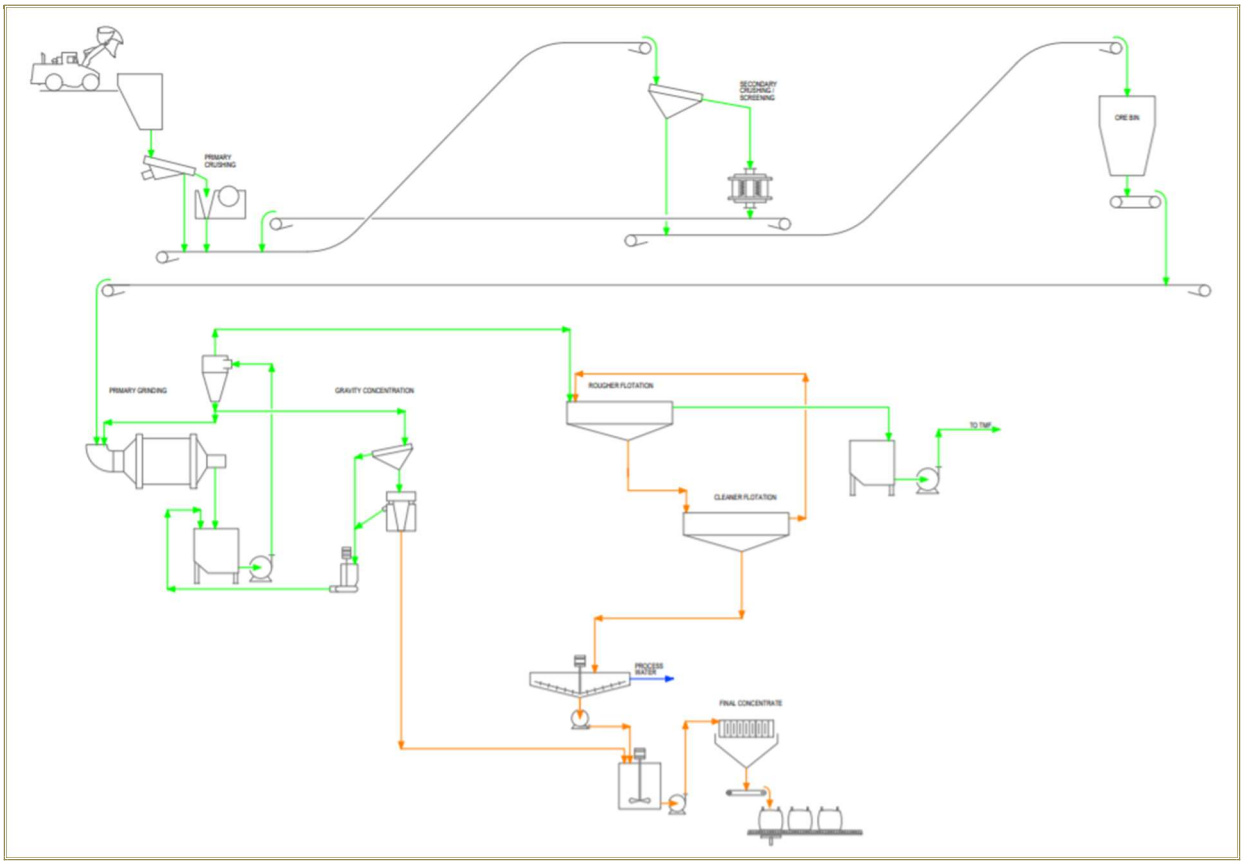
Criteria	Unit	Nominal Value	Source
General			
Plant Throughput	t/d	900	2020 mine plan
Crushing Plant Availability	%	50	Design consideration
Crushing Plant Throughput	t/h	75	Engineering calculation
Process Plant Availability	%	90	Industry standard
Process Plant Throughput	t/h	42	Engineering calculation
LOM Average Au Head Grade	g/t	5.26	2020 mine plan
LOM Average Ag Head Grade	g/t	8.77	2020 mine plan

Criteria	Unit	Nominal Value	Source
Gravity Au Recovery	%	46	G&T KM2121 and KM2522
Concentrate Au Grade	g/t	276	G&T KM2121 and KM2522
Overall Au Recovery	%	95.4	G&T KM2121 and KM2522
Overall Au Concentrate Grade	g/t	114	G&T KM2121 and KM2522
Crushing			
Availability/Utilization	h/d	12	Design consideration
Number of Crushing Stages	-	2	Vendor recommended – two-stage crushing plant
Crushing System Product Size (P_{80})	mm	10	Vendor simulation - estimated based on a final product aperture screen size of 14 mm
Crushed Material Stockpile			
Surge Bin Capacity (live)	t	900	Design consideration
Grinding			
BMWi (maximum)	kWh/t	14.1	G&T KM2522
Grinding Mill Type	-	Ball Mill	Selected to achieve target product size
Mill Diameter	m	3.2	Design consideration
Mill Length	m	5.2	Design consideration
Installed Power	kW	746	Design consideration
Circulating Load	%	300	Industry standard
Final Product Size (P_{80})	µm	150	Design consideration
Gravity	-	XD20 or Equiv.	Vendor recommended
Flotation			
Rougher Flotation Time Scale-Up	-	2.5	Industry standard
Cleaner Flotation Time Scale-Up	-	4.0	Industry standard
Rougher Flotation			
Laboratory Retention Time	min	8	G&T KM2121 and KM2522
Installed Retention Time	min	20	Engineering calculation
Mass Pull	%	15–20	G&T KM2121 and KM2522
Rougher Flotation Cell Size	m^3	10	Designed to achieve retention time
Number of Rougher Flotation Cells	#	6	Designed to achieve retention time
Cleaner Flotation			
Laboratory Retention Time	min	10	G&T KM2121 and KM2522
Installed Retention Time	min	40	Engineering calculation
Mass Pull	%	5–7	G&T KM2121 and KM2522
Cleaner Flotation Cell Size	m^3	3	Designed to achieve retention time
Number of Cleaner Flotation Cells	#	5	Designed to achieve retention time
Concentrate Dewatering			
Thickener Type	-	High Rate	Industry standard
Thickener Loading Rate	$t/h/m^2$	0.4	Design consideration
Thickener Diameter	m	4	Vendor recommended
Filtration Type	-	Pressure	Industry standard
Final Concentrate Moisture Content	%	8	Design consideration

Source: K-Met (2020)

17.3 Plant Design

A summary of the process flowsheet is presented in Table 17-1.



Source: K-Met (2020)

Figure 17-1: Overall Process Flowsheet

17.4 Process Plant Description

17.4.1 Crushing

Mineralized material from the mining operations will feed a crushing plant that will consist of three stages of crushing. The plant will process 75 t/h of mineralized material, operate 16 hours per day, and produce a final product P_{80} of 10 mm.

Mineralized material will be stockpiled near the jaw crusher or direct dumped through a 500 mm static grizzly into a dump pocket. Stockpiled mineralized material will be fed into the crusher by a front-end loader. The mineralized material will discharge into a feed hopper. Oversize mineralized material from the static grizzly will be removed for later size reduction using a rock breaker.

The mineralized material from the hopper will be withdrawn by a grizzly feeder with the oversize feeding directly into a 30 x 42 jaw crusher with an installed power of 110 kW. The primary crusher has a closed sized setting (CSS) of 101 mm.

The Screen Feed Conveyor will collect crushed product from both stages of crushing and feed a 1.8 m x 6 m double-deck vibrating screen. The +14 mm mineralized material will be conveyed to the secondary crusher. The -14 mm final product, at an estimated P_{80} of 10 mm, will discharge onto a conveyor and transferred to a surge bin.

17.4.1.1 Secondary Crushing

Mineralized material from the secondary crusher feed conveyor will discharge into a cone crusher with an installed power of 160 kW. The secondary crusher will reduce the mineralized material to a nominal product P_{80} of approximately 42 mm using a CSS of 44 mm. Crushed product will be transferred to the screen feed conveyor and circulated back to the double-deck screen.

17.4.2 Crushed Material Surge Bin

The double-deck screen undersize, with a final P_{80} product size of 10 mm, will be conveyed to the surge bin. The bin will provide 900 tonnes, or 24 hours, of live storage capacity. A belt feeder, installed with variable frequency drives (VFD) to control the reclaim rate, will feed the grinding circuit at a rate of 42 t/h.

17.4.3 Grinding

The grinding circuit will consist of a 746-kW ball mill operating in reverse closed-circuit with a cluster of hydrocyclones. The grinding circuit will be able to process a nominal throughput of 42 t/h of fresh feed and produce a final product P_{80} of 150 µm.

The mill discharge will flow into the cyclone feed pump box and combine with the gravity concentrator tailings before being pumped up to a cluster of hydrocyclones for size classification. The coarse underflow will flow by gravity back to the ball mill for additional grinding and the gravity circuit, while the fine overflow, at a final product P_{80} of 150 µm, will report to the flotation circuit. The hydrocyclones have been designed for a 300% circulating load, with a portion of the feed sent to the gravity concentrator. Gravity concentrate will report to the concentrate thickener, and the gravity tailings back into the cyclone feed pump box.

17.4.4 Flotation

Cyclone overflow will flow by gravity to the rougher circuit, which will provide 20 minutes of residence time. Frother MIBC and sulphide collector PAX, at natural pH, will be added to aid in flotation.

The slurry will flow through a bank of six 10 m^3 rougher flotation cells operating in series. Rougher concentrate will be collected in a common launder and fed to a conical pump box. The concentrate will then be pumped to the cleaner circuit.

Product from the rougher circuit will feed one stage of cleaning. The cleaner flotation circuit includes a series of five 3 m^3 flotation cells. The cleaner concentrate will be collected in a common launder and pumped to the concentrate thickener.

The cleaner and gravity concentrates will report to a 4 m diameter high-rate thickener. The thickener overflow will be sent to the process water tank, while thickened concentrate will be pumped to a stock tank feeding a pressure filter for further dewatering. Final concentrate, at approximately 8% moisture, will be bagged and sold to a smelter for further processing into gold doré.

17.4.5 Tailings Management

Rougher and cleaner tailings will combine in the final tailings pump box, and pumped to a dewatering circuit where it is anticipated that the dewater tailings will be comingled with the non-mineralized rock in the Rock Management Storage Facility. Therefore, no TSF is included in this report.

17.4.6 Reagents Handling and Storage

Reagents consumed within the plant will be prepared on site and distributed via the reagent handling systems. These reagents include PAX, MIBC, and flocculant. All reagent areas will be bermed, with sump pumps to transfer spills to the final tailings pump box, except for the flocculant: flocculant spills will be returned to the flocculant storage tank. The reagents will be mixed, stored, and then delivered to the flotation and dewatering circuits. Dosages will be controlled by flow meters and manual control valves. The capacity of the storage tanks will be sized to handle one day of production.

Table 17-2 summarizes the reagents used in the plant and their estimated daily consumption rates. The table also includes other major process consumables.

Table 17-2: Reagents and Process Consumables

Description	Delivered Form	Daily Usage (kg/d)
PAX	25 kg bags (dry)	126
MIBC	161 kg barrel (liquid)	24
Flocculant	25 kg bags (dry)	27
Ball Mill Grinding Media – 75 mm chrome steel	1-tonne bags	725

Source: K-Met (2020)

17.4.7 Air Supply

An instrument and plant air system, with one plant air compressor and associated dryers, filters, and receivers, will be housed in a compressor room inside the plant building. Two 22.4 kW air blowers will be used to provide air to the flotation circuits.

17.4.8 Water Supply

The following water types will be used in the process plant:

Process Water—Overflow water from the concentrate will be used as process water, along with reclaimed water from the tailings material dewatering circuit. This water will be used predominantly in the grinding circuit to dilute slurry to the required densities.

Fresh Water—Fresh water for the process plant will be pumped from a fresh water supply. Fresh water will be used as reagent make-up water, gland water, gravity flush water, filter wash water, and process make-up water.

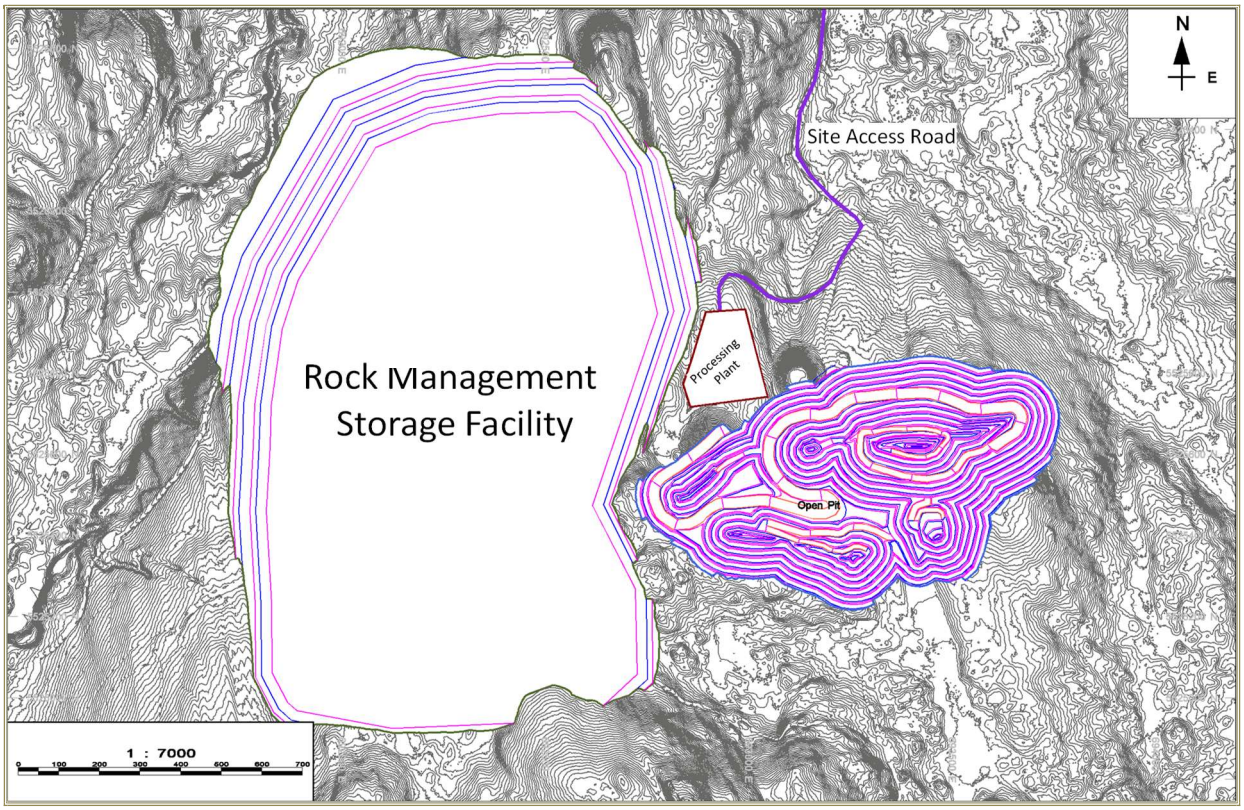
Reclaim Water—Water reclaimed from the tailings material dewatering circuit will be used as process water in the grinding and flotation circuits.

18 PROJECT INFRASTRUCTURE

18.1 Rock Management Storage Facilities

The RMSF is planned to be located immediately west of the final pit limits for the Elk deposit. Given the deposit configuration and extraction sequence, limited backfilling into previously mined-out areas has been planned for the Elk Gold Project.

The RMSF would be built in a series of 20 m lifts in a "bottom-up" approach, and the facility would be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with safety berms spaced at regular intervals, giving an overall operational slope of 2:1. The RMSF's total design capacity is 84 Mt.



Source: AKF Mining Services, August 2020.

Figure 18-1: Rock Management Storage Facility

18.2 Tailings Management Facility

The PEA mine plan envisages toll milling for the first three years of the mine life. Then in Year 4, a 900 t/d onsite mill will initiate mineral processing. It is anticipated that tailings material will be dewatered and comingled with non-mineralized rock in the RMSF and therefore no tailings management facility is included in this Technical Report.

18.3 Infrastructure

18.3.1 Road Access

The Elk Gold Project is accessed from BC Highway 97C, which is a four-lane all-season highway connecting Merritt to Peachland and Kelowna. The mine is reached via the Elkhart Road exit and driving 2 km south on the Golden Hills forest service road.

No additional site access roads are required for mine operations. The existing forest service road will be resurfaced and maintained for safe and efficient access to the Property for light vehicles and material shipments to and from the site.

18.3.2 Buildings

The buildings discussed below will be of modular design or consist of fully contained prefabricated components. They will be shipped to site either as complete units or will require minimal onsite construction, plumbing, and electrical work.

18.3.2.1 Maintenance Shop

Initial maintenance on mobile equipment will be carried out on a pad constructed of run-of-mine rock to create a stable foundation for maintenance activities. When the mine expands, a modular maintenance facility will be constructed to provide additional protection from the weather for maintenance work.

18.3.2.2 Hazardous Waste Storage Area

An area will be demarked as the hazardous waste storage area where waste oils, coolant, and other hazardous waste products will be storage in appropriate and certified storage containers before being shipped off site for the required disposal.

18.3.2.3 Emergency Response Building

The emergency response building will be a framed structure measuring 10 m by 20 m, and will house all the equipment and facilities to handle site emergencies.

18.3.2.4 Mineralized Material Sample Plant

A sample plant will be used for crushing and grinding assay samples taken from the plant feed. The sample plant will be located adjacent to the ore stockpile pad and will consist of a jaw crusher feeding a roller mill with an automatic sampler on the discharge belt. The sample plant will include all the guards required to protect employees from accessible conveyor belts and moving parts.

18.3.2.5 Warehousing and Storage

Warehousing and storage will be another pre-engineered, insulated sprung structure with a concrete slab on grade. It will measure 25 m by 30 m. This building will handle all inventories on site and house stock that must be protected from the environment. It will be associated with a lay-down yard for larger pieces of equipment and materials.

18.3.2.6 Sewage Treatment Plant

No sewage treatment plant is anticipated on site. Sewage from the on-site facilities will be collected by a contractor for disposal.

18.3.2.7 Fuel Storage

Substantial storage of fuel will not be required on site due to the easy access to the nearby highway. Some fuel storage will be required for the mine, haul, and light vehicle fleets, as well as for the heavy equipment and production of ANFO, a bulk explosive. Fuel storage will consist of certified containers in a bermed facility designed to comply with the Health Safety and Reclamation Code for Mines in British Columbia, and all other regulations.

18.3.2.8 Office Complex

It is estimated the mine site will have an office staff of 14 people. Office facilities will consist of modular office units to be certified constructed according to the BC Building Code and will be supplied with delivered potable water and generated electricity.

18.3.2.9 Mine Truck Shop

The truck shop will also be a framed structure measuring 55 m by 30 m, used to service the mining fleet for mine-specific operations. This building will also be equipped with overhead cranes, workbenches, and equipment, as well as areas allocated to permit maintenance and fabrication activities.

18.3.2.10 Truck Wash

An outdoor pad will be designated as the truck wash area where water will be collected into the site contact water system and water will be supplied from the pit dewatering pipelines.

18.3.2.11 Power

Single-phase electrical power exists along Highway 97C within 2 km of the Project location. There is potential to upgrade that line to three-phase power and extend the line to site to power milling and general power requirements. All power for the Project will be supplied by diesel generators.

19 MARKET STUDIES AND CONTRACTS

Certain information and statements contained in this section are "forward looking" in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and other parameters of the Project; mineral resource estimates; the cost and timing of any development of the Project; the proposed mine plan and mining methods; dilution and mining recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected LOM and other expected attributes of the Project; the NPV; capital; future metal prices; the Project location; the timing of the environmental assessment process; changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this Technical Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this report are subject to:

- There being no signification disruptions affecting the development and operation of the Project
- Exchange rate assumptions being approximately consistent with the assumptions in the report
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the report
- Labour and materials costs being approximately consistent with assumptions in this report
- Assumptions made in mineral resource estimates, including, but not limited to, geological interpretation, grades, metal price assumptions, metallurgical and mining recovery rates, geotechnical and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business, and economic conditions.

19.1 Market Studies

No Project-specific marketing studies were undertaken for this Technical Report. The planned gravity and bulk sulphide flotation will produce gold in a sulphide concentrate that is a fungible commodity for which an efficient global market exists. It is of high value density, meaning that the realized price of the contained gold is insensitive to the ultimate location of the customer and refinery, as freight costs are negligible in comparison to contained value.

Refinery terms of 97% payable gold in gold sulphide concentrate and a refining charge of $6.00/oz are based on the terms reported by nearby gold-mining operations. Silver is assumed to be 90% payable with a $0.50/oz refining charge.

19.2 Pricing

The three-year trailing gold price was used as the basis for pit optimization (Table 19-1). The three-year trailing gold price as of May 2020 was US$1,361/oz.

Table 19-1: 3-Year Trailing Average Gold Price (US$)

	Average	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
2020	1,629	1,560	1,597	1,591	1,682	1,716	-	-	-	-	-	--	
2019	1,392	1,291	1,320	1,300	1,286	1,283	1,359	1,412	1,498	1,511	1,494	1,470	1,476
2018	1,269	1,331	1,331	1,324	1,334	1,303	1,281	1,238	1,201	1,198	1,215	1,220	1,247
2017	1,273	-	-	-	-	-	1,260	1,236	1,282	1,314	1,279	1,282	1,261
3-Year Average	1,361	-	-	-	-	-	-	-	-	-	-	--	

The three-year trailing gold price is currently dis-aligned with various financial institutions' forward-looking gold price forecasts; therefore, the economic analysis in this PEA was carried out using a forward-looking gold and silver price curve starting in 2021 before reaching the long-term gold price forecast in 2025. The metal prices used are an average of seven financial institutions' price forecasts including Canaccord Genuity, TD Bank, Cormark Securities, Cantor Fitzgerald, Eight Capital, Haywood Securities, CIBC, and Scotiabank.

The forward-looking average price curves used in this economic analysis are detailed in Table 19-2.

Table 19-2: Metal Price Forecast Averages

Period	Gold Price (US$/oz)	Silver Price (US$/oz)
2021	1,776	19.27
2022	1,717	19.07
2023	1,644	18.52
2024	1,625	18.27
Long-term	1,600	18.11

19.3 Contracts

No contracts for the sale of the production have been entered into.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

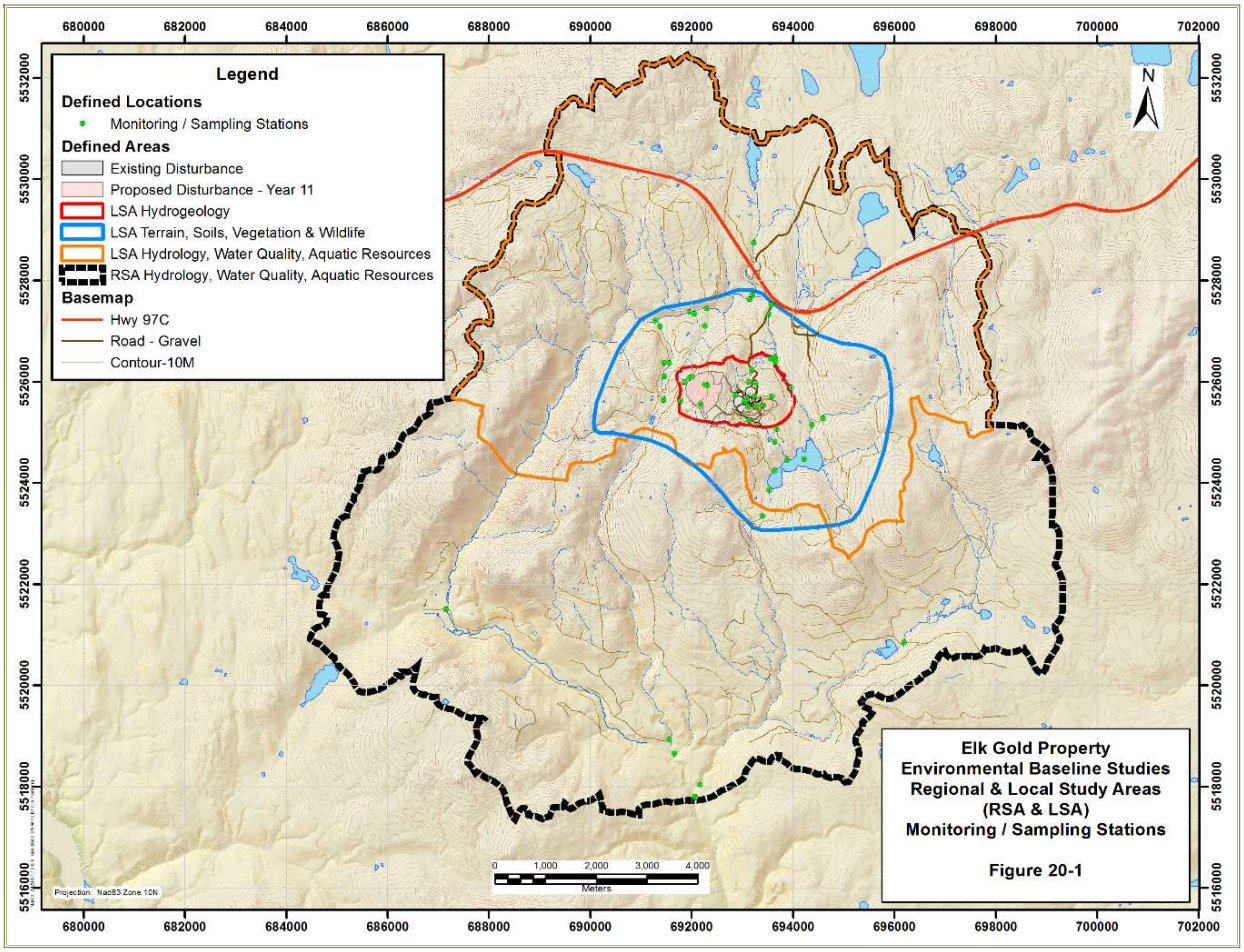
Information for this section has been sourced from individual baseline technical reports and from Chapter 2 of the Joint *Mines Act* and *Environmental Management Act* Permit Amendment Application (Bayshore, 2020). The author of this section did not participate in collecting or interpreting the data, or in writing up the reports and/or amendment application, but has reviewed all the reports and relevant sections of the amendment application to provide the following summaries. Palmer Environmental Consulting Group (Palmer) and its subcontractor EcoLogic Consultants Ltd. (EcoLogic) employ R.P.Bio and P.Geo/P.Eng. scientists and engineers registered in BC who have approved, signed, and sealed each of the reports that were reviewed.

20.1 Baseline Environmental Studies

Baseline environmental studies conducted at the Elk Gold Project have included:

- Meteorology and climate
- Soil and rock geochemical characterization
- Water quantity
- Water quality
- Sediment quality
- Fisheries and aquatic resources
- Ecosystems and wildlife
- Archaeology
- Cultural use.

A compilation map of the regional and local study areas (LSA) and sampling/monitoring stations is included as Figure 20-1. Summaries of these studies are discussed in separate sections below.



Source: Ecologic Environmental Consultants, May 2020

Figure 20-1: Regional and Local Study Areas Monitoring/Sampling Stations

20.1.1 Meteorology and Climate

To date, no weather station producing useable data has operated on the Property. However, long-term meteorology data sets are publicly available for several nearby weather stations operated by various government agencies and BC Hydro. Available climate data collected at these stations, along with detailed climate baseline information, were presented in *Elk Gold Climate and Hydrology Baseline Report* Palmer (2020a). Information presented in this section is drawn from Palmer (2020a) and *Elk Gold Project Mines Act Permit Amendment 199* (Bayshore, 2020).

Long-term periods of record were used to characterize the average annual and interannual variability expected at the Project. Station metadata collection years range from 1953 to 2019, with the following parameters monitored:

- Air temperature

- Relative humidity
- Precipitation
- Snowpack
- Snowpack snow water equivalent
- Wind speed
- Wind direction
- Air pressure.

Air temperature data sets (1968–2019) were compiled from the Brenda Mines weather station, due to its similar elevation and proximity to the Property. The mean monthly temperature ranges from −6.9°C in December to 14.6°C in July. The relative humidity (RH) collected at the nearby Pennask Summit Ministry of Transport weather station (MOTI Station ID 29029) is lowest in summer, at 61%, and highest in winter, at 93%.

Precipitation data were obtained from nearby weather stations—Elkhart Lodge Highway 97 (ECCC Station ID 112K653) for year-round data, and Elkhart Lake (MOTI Station ID 20292—for seasonal data. These data sets were used in conjunction with the Brenda Mines data sets to develop a synthetic long-term precipitation data set to characterize the conditions at the Property. The long-term mean annual average precipitation is 719 mm, with a mean annual runoff of 240 mm, and precipitation throughout the year. August and September are the driest months, and November through January the wettest. A frequency analysis method (Log Pearson III probability distribution) was used to calculate the return period precipitation for the Property area. The 1:200-year annual minimum and maximum precipitation levels are 481 mm and 903 mm, respectively, with 708 mm as the median 1:2-year return period.

The snowpack data at the MOE Brenda Mines snow pillow station were used as a proxy for the Project area. Snow begins to fall in October, with peak snowpack in late March to early April, and the snowpack lasts until mid May. The maximum annual snowpack at Brenda Mines ranges from a minimum 343 mm snow water equivalent (SWE) during a 1:2 return period, to a maximum 650 mm SWE in a 1:200 return period.

The annual potential evapotranspiration (PET) calculated from long-term temperature and precipitation data sets is 550 mm for the 1981–2010 climate normals period.

Hourly wind data collected from the Brenda Mines station (12 complete years) and the Pennask Station (10 complete years) show that the wind generally blows in NW or SE directions, with a maximum gust speed of 18.3 m/s recorded in November 2000. The average air pressure is lowest in winter, at 822 mbar, and highest in summer, at 829 mbar.

Median values for climate change models for the BC Southern Interior predict an annual temperature increase of 2.8°C and precipitation increase of 9% by 2080 over those of the baseline period of 1961 to 1990. This suggests that the Project area will have an increase in the mean annual temperature of

between 2.9°C and 5.7°C, and a mean annual precipitation increase of 62 mm, to a total of 756 mm over the 1961–1990 climate normals period.

20.1.2 Soil and Rock Geochemical Characterization

Information presented in this section is sourced from two reports: Terrain and Soils Baseline Report (EcoLogic, 2020a); and Metal Leaching and Acid Rock Drainage Characterization and Management Baseline Report (SRK, 2020).

20.1.2.1 Soil Characterization

EcoLogic (2020a) used terrain mapping and laboratory soil analysis methods to characterize the suitability of the Project area's soil for reclamation. The study found that the area soils have a fair suitability for reclamation, limited mainly by coarse fragment content and texture, with suitable salvage depths to a maximum of 50 cm. The soil nutrient regime within the mine footprint is characterized as poor to medium nutrient supply. The soils are loamy to sandy loam, and are slightly acidic at pH 5.6 to pH 6.0. Metal concentrations within the proposed mine footprint have an arsenic-in-soils exceedance above the Canadian Council of Ministers of the Environment (CCME) industrial guidelines. Other naturally elevated metal concentrations of lead, nickel, and copper above these guidelines occur away from the mining lease and disturbance area, and represent a naturally elevated background.

20.1.2.2 Rock Geochemical Characterization

SRK (2020) documents the geochemical characterization of non-mineralized hanging wall rock, mineralized material, pit walls, and overburden. Fieldwork was conducted in 2012 within the bulk sample pit areas that are at the heart of the proposed mine footprint. The data collected in 2012 were supplemented with more recent water quality monitoring data.

Samples were collected in three phases during 2012 for metal leaching and acid rock drainage (ML/ARD) testing from overburden, core representing non-mineralized rock distal to the veins, plus in-pit grab samples of non-mineralized footwall rock, pyritic rock immediately footwall to the veins, mineralized quartz veins and non-mineralized hanging wall rock. Seep water samples were also collected from the existing Rock Management Storage Facilities (RMSF).

Twenty-nine samples were tested at ALS for: total sulphur, sulphate sulphur, neutralization potential, paste pH, total carbonate analysis, and ICP element scan. Forty-six subsequent rock and overburden samples were tested at Maxxam and ACME as follows: total sulphur, sulphate sulphur, modified neutralization potential, paste pH and conductivity, total carbonate analysis, low-level mercury, total barium; and inductively couple plasma (ICP) element scan. Mineralogical characterization was performed on samples collected from the bulk sample pits by petrographic descriptions and quantitative (Rietveld) X-ray diffraction.

Leach tests using Meteoric Water Mobile Procedure (MWMP) methods were performed on four samples from historic mineralized rock piles, three samples from Pit 1 pyritic footwall, and three vein samples. Resulting leachate was collected and analyzed for a wide suite of parameters, as were seep samples.

From the results of these tests SRK determined that ML/ARD potential at the Elk Property is driven by the occurrence of sulphide minerals associated with auriferous quartz veins and immediate pyritic footwall rock due to the absence of readily reactive acid-neutralizing carbonate minerals. The host rock was classified as 90% non-potentially acid-generating (PAG), with less than 5% having a sulphide content greater than 0.2%. The vein material was found to be strongly ARD-generating almost immediately after exposure. SRK also concluded that non-mineralized rock placed in the proposed RSMF will react similarly to the existing RMSFs, which were Non-PAG after 20 years of exposure.

SRK proposed a ML/ARD management plan that included either shipping PAG vein material off-site for processing soon after mining or constructing a lined storage pad able to collect drainage for treatment and mining the footwall back to non-PAG host rock.

20.1.2.3 Geochemical Source Term Concentrations

Geochemical source term concentrations for non-mineralized rock dumps, pit walls, and mineralized material stockpiles were calculated by SRK for contact waters emanating from these facilities prior to dilution by local runoff waters. The predictions used scale-up calculations and geochemical modelling based on existing (historical) water-chemistry monitoring data and laboratory leach data. These inputs were used to generate mass loading rates from the existing footprint, as well as mass loading/concentrations from proposed mineralized material stockpiles and pit footwalls.

For non-mineralized rock dumps, the East Rock Storage Facility (ERSF) was used as an average test case. Historical water-quality monitoring downstream on Bullion Creek from 1991 to 2011 (prior to Pit 2 mining), together with the creek's flow rates, allowed loading calculations from the ERSF and Pit 1. Sulphate was chosen as the starting point for a conceptual source of heavy trace elements, as it remains in solution at relatively high concentrations, and is relatively insensitive to oxidation-reduction conditions. The sulphate release rate formed the basis of raw trace element concentrate calculations. The resulting annual sulphate release rates were 0.02 mg/kg for the ERSF, and 0.13 mg/kg/week for Bullion Creek. The difference in rates is because Bullion Creek is influenced by Pit 1, and the ERSF has a lower sulphur content in the non-mineralized rock. The sulphate concentrations were lower during spring freshet due to melt runoff.

Ore source terms were calculated by scaling up the MWMP laboratory leaching rates on mineralized material samples collected from 1995 mineralized material stockpiles. Average and worst-case water contact concentrations were calculated for multiple elements, using samples with a median pH value and lowest pH, respectively. The difference in concentrations was often three to four orders of magnitude between the two cases.

Source terms for pit walls were calculated in a similar manner as the non-mineralized rock, assuming all pyritic material is removed during mining, save for minor unrecoverable veins, and a 10 cm thick reacting zone. Again, there was a one to three order of magnitude difference between the best- and worst-case estimates.

No source term was calculated for overburden, as it does not contain mineralized material. It is assumed that overburden runoff will have concentrations similar to natural runoff.

A blasting residuals source term was calculated using a conservative powder factor of 0.3 kg/t (0.25 kg/t was used in mining Pit 2), and a planned mixture of 50% emulsion and 50% ammonium nitrate-fuel oil (ANFO). Most of the residual nitrogen post-blast will report to the non-mineralized rock, with a lesser amount (1%–10%) going to the mineralized material, and about 3% conservatively forecast to remain in the pit water. The loading rates for ammonia, nitrate, and nitrite from mineralized material and non-mineralized rock is calculated to range from 0.005 kg/d to 7.7 kg/d.

During Pit 2 bulk sampling, pre-strip soils and subsoils were not found to contain abundant mineralized material. Drainage from the proposed overburden stockpile is therefore expected to have similar characteristics as natural runoff waters.

20.1.2.4 Discussion

The intrusive non-mineralized hanging wall rock contains virtually no sulphide, and the proposed mining removes sulphide-bearing vein material through to the unmineralized footwall to ensure that both non-mineralized storage piles and pit walls will be non-acid-generating (NAG). The exception will be minor amounts of non-mineralized rock containing small, mineralized quartz veins that are non-recoverable. Pit walls will be composed mainly of non-mineralized intrusive rock, with occasional very thin and non-recoverable mineralized veins, such that footwall and pit wall surfaces will contain <0.1% sulphide and not be acid producing. Supporting evidence that the hanging wall will be NAG is that the East and West Rock Storage Piles are not acid producing more than 25 years after mining concluded in Pit 1. No significant environmental effects are therefore expected from RMSF runoff. Standard monitoring procedures include collecting and testing seepage waters from a sump constructed at the toe of the storage pile, and preventive mitigation measures include trenching the storage pile perimeter to collect and divert runoff seepage to the toe sump.

Mineralized material stockpiles are expected to be PAG, being enriched in pyrite and deficient in acid-neutralizing minerals, resulting in the potential for ARD. Onset of acidification would be rapid in summer and only slightly delayed during winter. Minor amounts (<1% combined) of arsenopyrite, chalcopyrite, sphalerite, galena, and tetrahedrite associated with the quartz-pyrite host could result in the leaching of trace amounts of arsenic, copper, zinc, lead, and silver. Cadmium is also present in mineralized material samples, and is strongly correlated with zinc. Trace amounts of antimony, arsenic, cobalt, manganese, and nickel have been found in mineralized vein material, but only zinc and cadmium were detected in pit wall seepage. Suggested mitigations include mineralized material storage under cover, frequent

mineralized material removals from site, and a lined mineralized-vein storage pad with the ability to treat drainage waters. Mercury is not thought to be of concern, as most samples had undetectable levels of mercury (<0.01 mg/kg).

20.1.3 Water Quantity

A hydrology baseline study initially conducted by Knight Piésold Consulting (Knight Piésold) in 2012 was updated in 2017, incorporating additional data collected between 2013 and 2016 (Knight Piésold, 2017). Additional work by Palmer in 2019 is documented in a climate and hydrology baseline report (Palmer, 2020a). Hydrogeological data were collected in 2012, 2017, and 2019 by Core6 Environmental (Core6) and Palmer (Palmer, 2020b). This section summarizes the results from those studies.

20.1.3.1 Hydrology

The proposed mine area is at the height of land of a gently undulating and glacially impacted plateau. All drainage waters from proposed open pit mining operations will flow either to Bullion Creek or from the proposed RMSF to Don Creek. Both Bullion Creek and Don Creek flow into Siwash Creek.

Siwash Creek was monitored by continuous stage data over five open-water seasons, and supplemented by at least 13 open-water stage-discharge measurements that allowed the development of stage discharge rating curves for each monitoring station. A long-term synthetic monthly streamflow series was developed for Siwash Creek using Water Survey of Canada (WSC) data collected in lower Siwash Creek. Regional and local discharge data were correlated using seasonal frequency paired regression, and the subsequent equations were applied to long-term WSC data. Two of the local measurement stations were recommissioned in 2019, and new ratings curves will be developed using data collected in 2019 and 2020.

Palmer (2020a) developed a streamflow model for upper Siwash Creek that used existing climate and streamflow data. Snowpack and snowmelt were calculated by a temperature-index method. Each agreed with measured values at nearby stations and were combined with modelled rainfall events to produce peak and low flows for several return periods. At station SC4, the peak instantaneous flow varied from 2 m^3/s for a 1:2 return period to 13 m^3/s for a 1:200 return period. The modelled low flow at SC4 was predicted to have no flow over a 7-day duration in a 10-year return period.

Potential impacts to surface water volumes include excess water input to Bullion and Siwash Creeks during dewatering, and changes to the normal water supply due to pit construction and in-pit seepage. The normal water supply to surrounding creeks would not be altered significantly as the proposed pit is at the height of land and normal pit sump-pumping would contribute equal amounts of surface flow. Post mining the flow could be diminished as in-pit seepage to groundwater and evaporation replaces surface flows. Don Creek impacts are more dramatic, as water from a constructed wetland is routed by pipe directly to Siwash Creek.

Mitigation measures that could be employed during dewatering would be to pump water over freshet when discharge would constitute a small percentage of the total flow. Post mining, the surface flows may

be altered depending on the loss of pit overflow to groundwater and evaporation, but the environmental effect of this change is considered neutral, with little residual effect to the local and regional study area. Don Creek is home to limited aquatic resources and the flows into Siwash are minimally impacted, so the environmental effect is also considered neutral.

20.1.3.2 Hydrogeology

Hydrogeological testwork consisted of six boreholes drilled and completed as monitoring wells, two of which were nested monitoring wells (Palmer, 2020b). The wells were drilled both up- and down-gradient of the proposed pit area to investigate groundwater levels, flow, and quality. The wells tested both shallow and deeper bedrock to determine if a contaminant plume was emanating from the pit and RMSFs, and to evaluate background water quality in the intrusive and volcanic bedrock.

Three hydrostratigraphic units were identified: topsoil, Nicola Group volcanic rocks, and Osprey Lake batholithic rocks. Hydraulic conductivity tests were conducted in the boreholes using rising- and falling-head methods. The hydraulic conductivity for Osprey Lake quartz-monzonite rocks ranges from 1.1×10^{-6} to 1.2×10^{-8}, while Nicola Group volcanic rocks were a consistent 4.2×10^{-8}. Groundwater flow was determined to reflect topography, with flow moving from higher to lower elevations, and ultimate discharge into Siwash Creek. In general, the groundwater movement uses shallow and deep bedrock fractures as the main conduits, with the greatest flow occurring in the upper weathered/fractured bedrock and overburden. Groundwater flow velocity in the proposed pit area is expected to be 0.02 m/d, while in the new RMSF area flows are expected to range from 0.5 m/d to 1.7 m/d.

20.1.4 Water Quality

20.1.4.1 Surface Water Quality

A surface water-quality baseline study completed in 2017 and updated in 2019 (Palmer)characterizes water chemistry and assesses the water quality for the years 1991 to 2014, and 2014 to 2019. Palmer produced a report, *Elk Gold Project—Surface Water Baseline Report* (Palmer, 2020c), that describes the results of sampling during those periods before and after issuance of the Effluent Discharge Permit (EDP) (Permit #106262). Information presented in this section is drawn from that report and Bayshore (2020), which identifies potential adverse effects from the Project so that comprehensive management plans could be developed to mitigate any effects.

The MOE issued the EDP under the BC *Environmental Management Act,* and sets conditions for discharge of seepage water from bulk sample mining operations. Included in those conditions is a requirement to collect surface water from certain sample stations three times a year and submit those samples to an approved laboratory for analysis of a given set of analytes. For the 2019 study, additional sample stations were selected, based on the geographic area reasonably expected to be affected by mine components. Bayshore used an independent water technician (G2O Services) to collect and submit samples to Caro Analytical Services' (Caro) laboratory in Kelowna. Samples were collected during freshet, and twice during

low flow conditions, as per EDP conditions, and additionally at monthly intervals during the open-water season.

Surface waters in the immediate mine area are classified as hard, in contrast to all other waters in the larger Project area, which are soft. All waters tested are neutral pH to slightly basic, and all stations show seasonal variability of analyte concentrations, with lowest concentrations during spring melt runoff. Analyte concentrations were elevated in the mine site area for many analytes, and those that historically exceeded water quality guidelines (WQG) for aquatic life included sulphate and fluoride, and the metals aluminum, cadmium, copper, iron, manganese, and zinc, with less frequent exceedances for arsenic, cobalt, thallium, and uranium. Visual examination of temporal trends at mine site locations indicated decreasing concentrations for aluminum, iron, and cadmium, and an increasing trend for sulphate.

There were few observed downstream effects on water quality. Of those analytes elevated in site waters, only sulphate was detectable at below guideline levels downstream of the discharge from Bullion Creek.

Cadmium provides an example of the distribution of contaminants of potential concern (COPC) within the study area. The concentration of cadmium has been up to 30 times the WQG in Pit 1 and Pit 2, although the concentration has decreased more than threefold in Pit 2 in the span of five years. Downstream, at Site 10 in Bullion Creek and station SC3 in Siwash Creek, the cadmium concentrations remain below WQG, and are comparable to reference sites EC1, SC4, and Site 13.

Aluminum and iron, in contrast, have been elevated in Siwash Creek both upstream and downstream of the main mine site discharge, but not in mine site waters. These data indicate that there were currently few measurable effects of mine site discharge downstream in Siwash Creek. During dewatering operations only Bullion Creek would be affected by increased Zn and Cd loading, and this would not be seen just downstream on Siwash Creek at station SC3. During operations there may be occasional spikes in cadmium and copper, plus nitrate residues from blasting seen in Don and Bullion Creeks but these do not carry further into Siwash Creek.

Dust and erosion causing sedimentation was examined as a potential environmental effect to surface water quality. Mitigation measures include management plans that limit the source of sediments, diverting clean water around the mine area, wetting roadways, constructing settling ponds, and using of (constructed) wetlands to remove sediment from mine discharge waters. With the use of these mitigations it was found that sedimentation had no residual effects on water quality.

20.1.4.2 Groundwater Quality Study

Groundwater quality studies were started in 2012, with Core6 completing five monitoring wells surrounding the mine footprint that tested both shallow and deep groundwater circulation systems. Palmer continued the study, collecting samples in 2017 and twice in 2019, and producing a groundwater report (Palmer 2020b), the results of which are summarized here.

Hydrostatic units were identified as overburden, Nicola Group volcanic rocks and Osprey Lake granitic rocks. The proposed mine area is near the height of land, so the primary surface recharge sources are snowmelt and rainfall. Groundwater facies were determined at each of the wells, using piper plots of major cation and anion concentrations. Seasonal and temporal variations in hydrochemical facies are minimal, with predominant groundwater types being (Ca + Mg) Co_3 and (Na + K) HCO_3, and are not affected either by depth or underlying geology: that is, shallow and deep groundwaters are hydraulically connected, and groundwater recharge may reach 50 m below ground surface. It is thought that secondary (carbonate) alteration and mineralization have a strong influence on groundwater chemistry, as geological mapping determined that Ca^{2+}, Na^{+}, and CO_3^{2-} were the predominant ions, indicative of extensive carbonate alteration.

Shallow groundwater flow from the proposed RMSF is towards the Don Creek watershed, with deeper groundwater flow limited by the low hydraulic conductivities of the deeper bedrock. The impact to Don Creek is expected to be low, since the non-mineralized rock will contain limited mineralized vein material. The proposed pit lies near the groundwater divide, with flows south of the pit being towards Gold Creek, with the bulk of the pit-influenced flow being to the north-northeast towards Siwash Creek. Low bedrock hydraulic conductivity suggests that the bulk of groundwater flow will be in the near-surface fractured environment, with only limited movement in the deeper bedrock.

Since near-pit groundwater quality could be affected by historical (Pit 1 and Pit 2) mining, monitoring wells were completed immediately down-gradient of the pits and at reference sites outside of pit influence. By comparing groundwater chemistry in potentially affected wells with blank (uninfluenced) monitoring wells, any natural above-guideline values can be used to distinguish above-guideline analytes that are mine related. This comparison can be used to assess future effects assessment of mine development on groundwater chemistry.

A full suite of both field-tested and laboratory-determined parameters and analytes were analyzed from each of four sampling events. Results reported included major ion chemistry, physical and chemical parameters, nutrients, and dissolved metals. All samples were analyzed by Caro in Kelowna, and results were compared against CCME guidelines for Freshwater Aquatic Life, BC Guidelines for Protection of Aquatic Life and Wildlife, and BC guidelines for Drinking Water Supply. Based on the deposit geology and past surface-water sampling, the primary groundwater Contaminants of Concern (COC) are sulphate, copper, iron, and aluminum. Secondary COCs are fluoride, cobalt, zinc, cadmium, manganese, and uranium. Background monitoring-well exceedances are mimicked by the downgradient monitoring wells, indicating that there are no adverse effects to groundwater quality from the historical mining activities.

Overall, the groundwater chemistry at the Elk Gold Project LSA is good, and generally meets guideline criteria. Elevated (above guideline) and naturally occurring concentrations of fluoride, total phosphorous, and several dissolved and total metals were identified at background monitoring well locations. No significant difference or increase in concentrations were measured in mine (downgradient) monitoring wells relative to background monitors. This indicates that no adverse effects to groundwater quality by historical mining activities were identified in the monitoring wells.

20.1.5 Sediment Quality

Sediment quality sampling was conducted by Palmer in 2011, 2015, and 2019, the results of which are reported in Palmer (2020c) and summarized here. Sediment samples were collected from multiple creek sites in 2012 (11 sites), 2015 (5 sites), and 2019 (6 sites), and analyzed for metals, total organic carbon, and nutrients by either ALS (2012) or Caro (2015 and 2019) using industry-accepted methods and QA/QC procedures. Two or three replicates of the finest sediment available were collected using standard protocols and placed in laboratory-supplied glass sample jars. Sampling was completed at water-quality and biota-sampling sites, proceeding in a downstream-to-upstream direction. The sampling program followed the BC MOE QA/QC protocols as outlined in their *Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators*.

Sediments were analyzed for 11 metals in 2012 and 2015, and for 37 metals in 2019, of which only eight have associated guidelines. Sediment polycyclic aromatic hydrocarbons (PAHs) and inorganic metals chemistry data were compared to two CCME *Sediment Quality Guidelines for the Protection of Aquatic Life*: Interim Sediment Quality Guidelines (ISQG) and Probable Effects Level (PEL). Sample results were also compared against the MOE's *Working Sediment Quality Guidelines for Aquatic Life, Wildlife, and Agriculture* (and amendments).

PAH analysis for all samples was below detection limits. Sample results exceeded the sediment quality guidelines for multiple metals at all sites, though not in all years, including at reference sites. That stations outside of historical mining influence had sediments above guideline levels suggests naturally elevated levels of metals occurring in the regional area.

Metals above CCME ISQG include arsenic, cadmium, chromium, copper, lead, and zinc. Mercury was below guidelines at all stations in all years, except once at one station, and the overage was not repeated in any subsequent year. Selenium, which was analyzed only in 2019, was well below guidelines at all stations. Copper was only above guidelines on Elkhart Creek and on Siwash Creek below the Elkhart confluence, suggesting influence from upstream Elkhart sources. All samples rarely exceeded CCME PEL guidelines, and those that did were also downstream of the Elkhart Creek confluence. The one exception to this is a sample collected just downstream of the Bullion Creek confluence where lead exceeded the CCME PEL guideline in one of three sample events. Samples collected from Bullion Creek were just slightly above CCME ISQG guideline for lead.

20.1.6 Fisheries and Aquatic Resources

Palmer completed field fisheries and aquatic resources studies in 2012, 2015, 2017, and 2019 at five sites along Siwash Creek, four sites within Siwash watershed tributaries, one site in Siwash Lake, and two reference sites in Elkhart Creek and Galena Creek. Study results were compiled and presented in a *Fisheries and Aquatic Resources Baseline Report* (Palmer, 2020c) the findings of which are summarized here.

20.1.6.1 Stream Site Studies

Stream site studies included: periphyton and benthic invertebrate community composition and abundance; benthic invertebrate tissue chemistry; fish habitat; and fish abundance, condition, diet, and tissue chemistry. The Siwash Lake study examined physical limnology, and community composition for phytoplankton and zooplankton (lakes). Both stream and lake studies used standard and approved field, laboratory, and data methodologies. Appropriate QA/QC checks were used during the collection, analysis, and results interpretation of all fisheries and aquatics samples.

Periphyton is an assemblage of algae, bacteria, fungi, microinvertebrates, bacteria, and detritus found attached to aquatic substrates such as rocks. The measurement of periphyton biomass and associated chlorophyll *a* concentrations provides a mechanism for assessing ecosystem health and primary productivity in a watershed. Periphyton sampling was conducted at seven sites in 2012, and six sites in 2019, two of which were common to both sampling events. Five of the sites were control stations that have had either no impact or no direct impact from historical mining. Two sites are within watersheds impacted by historical mining, two sites may be impacted by proposed mining, and two are further downstream of impacted areas.

In Siwash Creek and associated tributaries, periphyton density ranged from 0.56 mg/cm^2 to 3.52 mg/cm^2, and taxa richness varied from 22 to 36 taxa, with diatoms and cyanobacteria as the dominant taxa. There was almost no between-site variability in diversity or evenness. For all periphyton community metrics, there were no consistent differences between reference sites and exposure sites downstream of the mine site. This suggests that mine discharges have had no effect on the periphyton community at sites in Siwash Creek and associated tributaries downstream of the mine site.

Benthic invertebrate community sampling allows a view of the aquatic health, and potential long-term impacts, at specific aquatic sites. Benthic abundance and variability are two measures useful in determining aquatic health. High benthic abundance is found within productive systems, and low abundance is more typical of a river system experiencing toxicity or habitat disturbance. The Canadian Aquatic Biomonitoring Network (CABIN) protocol was used in benthic sampling.

Community abundance at the reference station had the lowest abundance relative to mine-impacted stations, suggesting that discharge from historical mining has had no measurable effect on the downstream receiving environment. Benthic variability between sites and between years was high, but the dominant order at all sites were Diptera or Ephemeroptera, with the pair composing a relative abundance of 50% to 90%. Metal concentration-sensitive orders Tricohopera and/or Plecoptera were found at most sites except Gold Creek, where high iron has been found since sampling began. This is indicative of relatively healthy invertebrate communities in all streams but Gold Creek. A review of the Bray–Curtis Species Dissimilarity Index of the reference site to sites downstream of mine influence indicates that mining activity to date has not measurably affected the benthic invertebrate community structure.

Taxa richness, Ephemeroptera, Plecoptera, Trichoptera (EPT) richness, diversity, and evenness tests were performed on the benthic invertebrate data as a further test of overall aquatic environment health. The benthic invertebrate community at the Project showed high ratings in all tests with no temporal trends, a consistent trait since the first benthic surveys in 2004, 2007, and 2008. This is indicative of a healthy ecosystem with high habitat complexity and stability. A Shannon–Weiner diversity test had lower diversity numbers than the Bray–Curtis and Simpson's diversity tests but was in agreement for the evenness test. The reason for this discrepancy is unclear.

CABIN analysis used the Okanagan 2017 reference model, and showed fewer recovered taxa than the model predicted. The observed to expected (O:E) ratio was exceeded at two sites (far downstream, and one of the two reference sites), and lower in the upper reaches of Siwash Creek, Bullion Creek, and Gold Creek, several sites of which are unaffected by mining to date. There is a suggestion that variant creek flows may have contributed to unequal sampling for benthic communities in these areas, as additional flows in the reference creek added to the downstream Siwash sample site.

Metal content in benthic invertebrate tissue was elevated for mercury at exposure sites over the reference site, suggesting that metal accumulation is occurring within the benthic invertebrate community. Although elevated, the metals concentrations for which there are tissue residue guidelines (mercury and selenium) were both below guideline levels.

Despite elevated metal content at exposure sites, the Property benthic invertebrate community has maintained high abundance, diversity, and richness since early site investigations, indicative of a healthy benthic ecosystem with high overall habitat complexity and a stable benthic community.

Biomonitoring of fish species is another method of monitoring watershed health, since fish are relatively long lived and have specific environmental requirements that may be sensitive to perturbations caused by industry. Fish and fish habitat studies conducted in 2012, 2017, and 2019 employed electrofishing, minnow trapping, and gillnetting. These methods assessed fish age, length, weight, diet, maturity, and tissue metals. Creek walking allowed the assessment of spawning, rearing, and overwintering habitats, as well as barriers to fish in reaches apparently devoid of fish populations. Data on fish habitats were collected from Siwash Creek and four of its tributaries: Gold Creek, Bullion Creek, Don Creek, and Agur Creek. Reference creeks Elkhart Creek and Galena Creek were also assessed.

20.1.6.2 Creek Assessments

In the study area, Siwash Creek was found have a low gradient (<3%), with irregular wandering patterns and overhanging vegetation and boulders, as well as large woody debris for cover in downstream reaches. Overall, the creek was assessed to have good rearing habitat, but limited areas for spawning and overwintering. In addition, low creek flows in summer impair fish movement and cause isolated pocket waters. Rainbow trout in the creek reach a maximum mature length of 200 mm, versus 300 mm for those living in Siwash Lake. Analysis of fish tissue for metal concentrations was performed on fish collected from all fish-bearing creeks, including reference creeks. All samples were found to have mercury levels

above guideline levels for the protection of wildlife, and below Health Canada guidelines for human consumption. This suggests that the Siwash Creek watershed has elevated natural mercury concentrations that are being bioaccumulated by fish. Selenium concentrations in fish tissue however, were well below BC guidelines.

Gold Creek is a high gradient stream (7%–18%), with a narrow mean wetted width (<1 m), that outlets into Siwash Lake. The stream flows though mature coniferous forest and clear-cut areas lacking a forested buffer. Several fish barriers over the bottom reach (100 m), together with shallow water depths, led to the conclusion that Gold Creek is likely non-fish bearing.

Bullion Creek has a shallow gradient (5%), and lower, shallow flow in comparison to other creeks in the study area. It frequently experiences dry conditions in summer, and strong flow velocities during freshet. Although it has limited cover, and no overwintering or spawning habitat, juvenile fish were collected in Bullion Creek near its confluence with Siwash Creek, and it appears the area is used for seasonal rearing. Fish migrating upstream under moderate flows would become stranded and desiccated as the stream dries up each year.

Don Creek is a narrow, shallow stream, and fish habitat is limited to 10 m upstream of its confluence with Siwash Creek, due to a 34% gradient barrier. Within the fish-bearing section, cover is provided by undercut banks and small woody debris. This section has no spawning or overwintering habitat for rainbow trout.

Agur Creek is physically similar to Don Creek, and provides only seasonal rearing habitat along its lowest reaches, near the confluence with Siwash Creek.

Elkhart Creek is a low-gradient (1.5%) reference creek that flows into Siwash Creek. Though it lacks forest canopy cover, boulder and overhanging vegetation made up to 20% cover. The creek is assessed to provide good rearing habitat, but lacks gravel-based spawning and deep-pool overwintering habitats.

Galena Creek is a low gradient (1%) far reference creek that feeds into Siwash Creek over 8 km downstream from the LSA. It sources a large watershed over its 11 km length, which takes an irregular wandering pattern through an unconfined valley. Cover (20%) is mainly boulder and overhanging vegetation, with between 1% and 20% low canopy cover. There were limited overwintering pools, and only minor gravel substrate for spawning, but good rearing habitat.

Fish populations were found in all streams but Gold Creek and Don Creek, with Siwash Creek, Galena Creek, and Elkhart Creek having the highest densities of fish populations. Fish were caught only in the first few metres of Agur Creek and Bullion Creek, though the latter may support upstream fish movement during freshet in May and June. For the rest of the year, Bullion Creek does not provide fish habitat, as it is either dry or contains shallow, stagnant pools.

Bayshore (2020) noted that while historical mining in 1992–1995 and again from 2012–2014 has had little apparent effect on aquatic resources, the proposed project has the potential to negatively affect the

aquatic environment, and ultimately to fish through changes in habitat and sedimentation, plus discharge of contaminated mine contact water. The study concluded that negative effects through these pathways during construction and operations can be mitigated by strict adherence to various management plans. These plans include an Environmental Management System, Surface Erosion Prevention and Sedimentation Control Plan, Mine Site Water Management Plan, and Vegetation Management Plan.

20.1.7 Ecosystems and Wildlife

Wildlife and wildlife habitat studies were completed on the Elk Property in 2012 and 2019. Information contained within this section is drawn from reports that present the data and interpretations, namely the *Vegetation and Wildlife Baseline Report* (EcoLogic, 2020b) and Chapter 2.10 of Bayshore (2020). Along with describing the local and regional ecosystems, the studies were designed to:

- Identify invasive plant species on the Property
- Inventory resident and migratory wildlife species
- Identify federal and provincial at-risk species of vegetation and wildlife that may be present on the Property
- Analyze soils and vegetation for trace elements (metals and metalloids)
- Assess potential adverse effects of mining activities on wildlife to allow development of management plans to avoid and/or mitigate the adverse effects.

20.1.7.1 Ecosystem and Wildlife Desktop Study

The baseline study identified federal or provincial legislation or guidance relevant to the vegetation or wildlife baseline study. These statutes and regulations (five for vegetation, and eight for wildlife) pertain to the conservation of vegetation, wildlife, and wildlife habitat, especially those that are or could be most threatened.

The Project area is divided into two study areas, namely the Regional Study Area (RSA) and the LSA. The RSA ecosystem is defined as that area surrounding the Property and lying above 1,200 masl, and encompasses roughly 298,600 ha. The area is almost entirely within the Thompson Okanagan Plateau ecoregion, though a small area to the south lies within the North Cascades ecoregion. Portions of five ecosections and eleven biogeoclimatic (BGC) units occur within the RSA, which was heavily glaciated, resulting in a rolling plateau blanketed by glacial-lacustrine sediments. The area has been extensively logged, leaving only a patchwork of native ecosystems, old-growth forests, wetlands, and riparian areas. Historical mining and mining exploration have added to the anthropogenic disturbances.

The LSA, at 1,884 ha, lies 1,455 masl and 1,733 masl, and occurs within the Western Okanagan Upland ecosection, Cascade Natural Resource District, and Hayes landscape unit. It is defined by watershed boundaries, and includes areas that will see both direct and indirect impacts of proposed mining. The LSA encompasses three BGC ecosystem units, being Montane Spruce Similkameen Very Dry Cool, Montane Spruce Cascade Dry Mild, and Engelmann Spruce–Subalpine Fir Thompson Very Dry Cold variants. The

area normally experiences cool, moist winters, and hot, dry summers, though storms from the Pacific can bring rain in summer, and snow in winter. Cold arctic fronts cause clear, cold conditions a few times each winter. Forests are dominated by Douglas fir (*Pseudotsuga menziesii*), white spruce (*Picea glauca*), Engelmann spruce (*Picea engelmanni*), subalpine fir (*Abies lasiocarpa*) and lodgepole pine (*Pinus contorta*).

An initial desktop assessment sought to identify potential vegetation, wildlife, and ecosystems at risk within the LSA using various data sources as outlined in Bayshore (2020). This assessment was used to guide searches during on-the-ground studies. Fieldwork in 2019 updated the 1:20,000 scale terrestrial ecosystem map (TEM) previously completed in 2012 to include more recent timber-harvesting disturbances, along with the 2012–2014 bulk sample mining operation. The new TEM was completed at a more detailed 1:5,000 scale, which allowed mapping of ecosystem rather than terrain polygons. Thirty visual checks were completed, representing 28% of all TEM polygons originally established by 56 field plots in 2012.

20.1.7.2 Ecosystem and Wildlife Field Study

The September 2019 field study sought to identify all at-risk red- and blue-listed wildlife and plant species and ecological communities potentially occurring within the LSA, along with invasive plant communities. Techniques used included use of trail cameras, bat detectors, visual encounter surveys, and wildlife sign transects.

Three ecosystems at risk were found to occupy 11% of the LSA, a fraction of which occurs within the mine footprint. Plant species-at-risk studies identified one red-listed and 12 blue-listed species that are likely to occur, though only one blue listed species was found within the LSA.

Invasive plant species were not the focus of the field study, and of the 16 regionally or provincially noxious species identified in the desktop study, only one (Canada thistle) is currently subject to yearly eradication efforts. A new small population of a noxious and highly invasive species (tansy ragwort) was found to occur within the LSA, and needs to be eradicated before it has a chance to spread.

Soils and vegetation metal analysis identified multiple sites where soils have naturally occurring elevated values above industrial guidelines for human health, including arsenic, copper, lead, and nickel. Vegetation analysis for metals identified three sites above guidelines for vegetation, with two situated next to access routes, and one occurring on the West Rock Storage Facility (WMSF). The study serves as a guide to future movement of reclamation soils.

Wildlife surveys included searches for amphibians and reptiles (toads and salamanders), birds (raptors, owls, and water birds), and mammals (including bats). The desktop study identified five species at risk for ground-truthing surveys. Field surveys detected amphibians at 6 of 13 wetlands, though fall conditions were not ideal, due to low temperatures leading to less activity. No threatened species of amphibians were found.

Thirty-seven bird species were identified during the field surveys, including one blue-listed northern goshawk (dead juvenile), and one red-listed Swainson's hawk (migratory species at this site). Raptors were found widespread on the Property, but at low densities. Persistent poor weather conditions hampered the survey. Automated bat detector surveys recorded 481 bat passes (2.3 passes per detector per night) from eight species groups. No owl species were documented on the Property during the survey.

Of the 29 mammals documented within the LSA in 2012 and 2019 using trail cameras and track-scat-browse-sound identification methods along transects, the most recorded mammals were deer, moose, red squirrel, and snowshoe hare. The only species-at-risk mammal confirmed on the Property was the little brown *myotis* (bat).

20.1.7.3 Discussion

Habitat analysis identified multiple habitat types within the LSA, including: open disturbed areas (6% of LSA), recent clear-cuts (<15 year, 40%), pine plantations (12%), wetlands (3%), riparian (11%), and forest (36%). The areas for each habitat type were estimated, and totalled 108%, likely due to overlapping identifications by multiple workers. The riparian forests and wetland habitats were found to offer the best mammal habitats through a combination of thermal cover, linear travel routes, forage, and security cover. Forested areas adjacent to riparian and wetland areas are also considered good habitat.

Freeform (2020) noted that the proposed project could create a linkage by way of ingestion between contaminants and terrestrial receptors though deposition of dust and/or discharge of effluent. It was also stated that loss of habitat or changes to ecosystems during construction and operations could negatively effect vegetation and wildlife populations though the Project is proposed in an already highly disturbed and fragmented landscape. It was concluded that adherence to several management plans would limit these negative effects. The plans included the Surface Erosion Prevention and Sediment Control Plan, Soil Management Plan, Construction Environment Management Plan, and Vegetation, Invasive Plant Management Plan, and Fuel Management and Spill Control Plan. It is stated within the report that mitigation measures outlined within the plans would reduce the residual effects to a low or negligible magnitude. It was also acknowledged that while the nature of the disturbance will be severe and permanent over part of the project area, it will occur over a relatively small spatial scale. Habitat deterioration during operations would occur over a 73-ha area around the Project infrastructure and a portion of this would be reversible post closure.

20.1.8 Archaeology

Two Archaeology Overview Assessments (AOA) were completed over the same general area within the LSA; one in 1992 for Fairfield Minerals, in preparation for bulk sample mining, and one in 2011 for GMMC prior to diamond drilling. Information for this section is drawn from those two reports (Rousseau, 1992; Gould, 2012). Results of more recent engagements with First Nations communities are documented in Freeform (2020).

The Project area falls within the asserted traditional territories of the Nlaka'pamux Nation and the Okanagan Nation, specifically their member communities the Westbank First Nation, Upper Nicola Band, and the Upper Similkameen Band. The overview studies were conducted to assess the archaeological site potential within the proposed impact zones associated with the then proposed developments. Information resulting from the studies determined for whether further archaeological studies were necessary, and if so to help formulate efficient strategies to conduct such studies. The 1992 archaeological study was the first to have been conducted in the Siwash Lake area, and examined the eastern half of the LSA. At the time, the only forestry clear-cuts were in the Discovery Zone area and the Siwash North proposed mine area.

Similar subsistence and seasonal settlement practices were followed by members of the North Okanagan and Nicola First Nations groups. Hunting and gathering formed the basis of food procurement for their communities, and the Project area may have been part of their traditional resource grounds. Rousseau and Gould both state that the highest potential ground for archaeological artifacts is on flat terrain within 100 m of a pond or lake. On that basis, the 1992 study found that there was low to nil chance of historical and prehistoric archaeological site potential in the mine area.

The 2011 AOA used a Geographical information system (GIS) desktop study approach to determining archaeological potential, with ground truth traverses in areas of highest potential. Archaeological site records (via Remote Access to Archaeological Data [RAAD]), and site-potential ratings (including ground slope, aspect, distance from water, BGC zone, ungulate winter range, and proximity to communities) were added to the GIS model as potential-rated polygons. The model was queried to determine areas of overlapping polygons with the highest potential, and the resulting plot formed the basis for field ground truthing. Like the 1992 study, the 2011 AOA determined that the area may have been used for seasonal hunting and gathering and travelling pursuits, and that the archaeological potential ranks low to moderate. The 2011 AOA team attempted to find one documented archaeological site (DIRb-5) located near the outlet of Siwash Lake was but it could not be located. A second potential shelter area under overhanging rocks was also examined without discovering cultural materials or pictographs. No other archaeological sites were discovered during the two-day inspection.

20.1.9 First Nations Engagement

GMMC has been engaged with First Nations communities around the project since 2011 when exploration and bulk sample mining were taking place. Since the acquisition of GMMC by Bayshore in 2019, Bayshore has provided project information to 26 First Nations bands and associations. Many of these groups simply wish to receive notification of GMMC's activities. Nine bands and associations are participating the Mine Review Committee related to the Joint Mine Permit Amendment Application submitted by Freeform in May 2020.

It is noted that engagement activities to date have focused on the 200 t/d mine plan which forms the basis of the mine permit application and that the expansion of the mine plan to the 900 t/d envisioned in Year 4

of this PEA, additional consultation with First Nations will be required as a part of an Environmental Assessment process.

Freeform is currently negotiating Impact Benefit Agreements with several bands in parallel to the mine permit consultation.

20.1.10 Cultural Use

The Elk Gold mining lease and exploration claims areas have multiple overlapping tenures or land use interests. These include First Nations' traditional territorial claims; forestry, energy, and mineral land tenures (e.g., placer); trapping; and range grazing licences. Recreational users, such as hunters and snowmobile and ATV riders, also have an interest in the vicinity of the mine footprint. Bayshore (2020) discusses these cultural uses and information contained in this section is drawn from that report.

The mining lease area does not overlap with any private land, provincial parks, protected areas, or agricultural zoning. There are three private property holder areas, 1.6 km and 2.0 km north, and 1.4 km south of the mining lease. The closest provincial parks are 13 km away. There are no ecological reserves overlapping the Elk Gold Project.

Potential Project impacts to First Nations communities include loss of land and traditional resource use, treaty rights infringement, and asserted First Nations rights and interest impacts. Bayshore is currently communicating with all affected First Nations groups in an effort to identify all real and perceived conflicts and to develop resolutions.

Aspen Planers has conducted several clear-cut operations on the Property, and other forest licences exist within the claims area. A wind farm for electrical generation was proposed on Property claims north of the mining lease. The property is within a provincially approved placer staking area, and placer claims have historically been staked over the mineral claims. No hunting or fishing outfitter territory is known in the area; one trapline overlaps the Project area. A grazing licence held by Lake Douglas Cattle Ranch overlaps the Project footprint, and the Project is within the Merritt timber supply area (TSA) administered by the Cascades Forest District. There are no water use licences overlapping the Project, and no downstream users within the RSA. There are two unlicensed groundwater wells located 1.6 km north of the mining lease.

Recreational ATV and snowmobile use have been documented on the lease and claims area, and hunters have regularly ignored warning signs that prohibit trespass. Over the past eight years, vandalism and theft have become commonplace on the Property, and GMMC has repaired damages to locked gates multiple times.

During construction, operation, and post closure reclamation the Project will affect traditional cultural use; however, the spatial area of disruption is relatively small in comparison to the surrounding area available for public use. The potential for spatial overlap with other projects in the area was found to be negligible hence the cumulative effects are considered to be insignificant.

20.1.11 Potential Environmental Impacts

Several receptors have been identified in previous sections within this chapter as being potentially affected by the Project, including periphyton and aquatic plants, aquatic invertebrates, and ultimately fish (rainbow trout). Vegetation, and mammals that feed upon the plants, are also potential receptors.

Within the LSA watershed, only Siwash Creek and the lower 10 m of Don Creek provide relevant fish receptor locations that could be affected by pollutants, heavy metals, or metalloids released into the aquatic environment. Groundwater also provides a vector for contaminated water, and although no wells currently exist in the mine area, groundwater protection for future development needs to be a priority. Careful management of mining non-mineralized products is therefore essential to maintaining healthy hydrologic and hydrogeologic environments.

Vegetation supported by various ecosystems on the Property which could potentially be impacted by the Project include old growth and new forests, wetlands, and riparian areas. Invasive plants introduced or allowed a foothold in disturbed areas could threaten local vegetation and therefore need to be controlled or eradicated.

Mammals that feed on vegetation and/or other animals could be negatively affected if soils are not adequately isolated from mine pollutants. Dangerous mine-created cliff edges will need to bermed for wildlife safety. Human activity and interactions with wildlife in the area, whether for employment, recreation, or cultural reasons, will need to be strictly controlled. Incidents that could potentially cause injuries or death due to unplanned interaction with heavy equipment or unprotected cliff edges will need to be closely monitored and mitigated.

Overall, robust management planning is needed to identify and mitigate these and other negative vectors. Working collaboratively with all potentially affected groups is desired to develop a Project plan that respects environmental and cultural needs.

20.1.12 Potential Project Expansion

Freeform has submitted a *Mines Act* permit amendment application for a 70,000 t/a mining permit from the existing 10,000 t/a M-199 permit. As submitted, the amended Project involves offsite toll milling and is below the British Columbia *Environmental Assessment Act* (BC EAA) Reviewable Projects Regulation threshold of <75,000 t/a and <50% expansion that would trigger an Environmental Assessment (EA).

This PEA envisages a change in mining technique from selective mining at 200 t/d to bulk mining at 900 t/d in Year 4. Such a change would push the Project above the BC EAA reviewable projects threshold and require an amendment resubmission that would trigger a more rigorous EA. GMMC will be required to apply for an amendment allowing for both the higher mining rate and onsite milling. Under such a scenario, additional environmental and engineering studies will be required, along with in-depth consultation with all interested parties. The anticipated cost of a reviewable permit amendment could be as high as $5 million.

20.2 Reclamation and Closure

A Reclamation and Closure Plan (RCP) was produced as a chapter in the Joint *Mines Act* and *Environmental Management Act* Permit Amendment Application (Bayshore, 2020) from which the following information was drawn.

The RCP is a written strategy that complies with the necessary standards of the *Mines Act* (1996) and the Health, Safety and Reclamation Code for Mines in British Columbia (2017) and the *Environmental Management Act* (2003). The Project closure objectives include: Design of mine for closure; achieve long-term physical stability; achieve long-term chemical stability; consider future use and aesthetics; and meet the desired end land use criteria.

Designing for mine closure involves identifying post reclamation forces that could affect mine closure components factoring these items into the mine design such that the reclamation objectives can be achieved. To professionally design for mine closure the mine development and operations planning should include planning for reclamation needs. All planning should be completed in consultation with others that have an interest in the land, the mine, and the environment. An important part of an RCP is the costing of reclamation and closure such that security can be established to ensure closure costs are met.

Reclamation and Closure costs were based upon the Regional Reclamation Bond Calculator (EMPR, 2020) which includes a separate 10% management cost and 20% contingency cost. Reclamation costs for the 2-year program are estimated to be $3.934 million.

The closure cost estimate assumes no long-term water treatment, and minimal water quality monitoring and does not present a 100-year NPV model. Following reclamation, closure was designed to be a 5-year process and costs are related to monitoring and maintenance activities. The 5-year cost of closure is estimated at $1.456 million.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

21.1.1 Owner's Costs

Owner's costs included in this economic analysis comprise payments payable to the Property vendor, technical studies associated with mining, environmental and socioeconomic aspects of the Project, permit and authorizations associated with the Project expansion. The total Owner's costs over the LOM is $16.2 million.

21.1.2 Mine Capital Cost

The mine capital cost for the Elk Gold Project is limited, as the primary production fleet of equipment is planned to be leased from the suppliers, and that leasing cost is built into the operating cost estimate. The blasthole drills, excavators, haul trucks, dozers, and graders are all planned to be leased. The leasing cost is applied to each period of the mine plan based on the number of units required, and as the units are leased, no replacement costs are including in the capital cost estimate.

The initial capital cost of the mine includes: the purchase of ancillary mobile equipment required to support the first three years of 70,000 t/a operations; the modular office and other ancillary facilities; and the site preparation prior to construction. The pre-construction activities are limited to excavating a water settling pond at the toe of the RMSF along with the associated water-collection and diversion ditches, as well as salvaging organic material, topsoil, and till from the initial footprint of the RMSF and first pit phase.

The expansion capital cost is associated with the increase in mining rate and includes additional ancillary equipment and expanding site facilities to support the larger operation.

The ancillary equipment includes light trucks and service vehicles, backhoes, and fuel trucks, along with a number of other required open pit mining support equipment. The mine capital cost is detailed in Table 21-1.

Table 21-1: Mine Capital Cost Summary

Item	Initial Capital ($ '000s)	Expansion Capital ($ '000s)	Total Cost ($ '000s)
Modular Office/Dry Facilities	100	50	150
Fuel Storage/Pumps	100	100	200
Explosives Magazines	40	75	115
Surface Water Management	250	300	550
RMSF Topsoil Salvage	100	250	350
Open Pit Topsoil Salvage	150	100	250
Access Road Resurfacing	20	-	20
Ancillary Mobile Equipment	1,100	1,350	2,450
Initial Pit Dewatering	85	-	85
Working Capital	5.000	-	5.000
Total Mine Capital	**6,945**	**2,225**	**9,170**

21.1.3 Process Capital Costs

The initial years of the mine plan envisage toll milling at existing off-site facilities and therefore have no planned capital expenditures.

The mill capital expense is scheduled for Year 3 of the mine plan when the 900 t/d mill will be constructed on site. This PEA-level capital cost estimate for a 900 t/d mill in south central British Columbia is $23.8 million, which includes the milling equipment, concrete, steel structures, engineering, procurement, and construction management (EPCM), and contingency. This value was derived from the cost of similar-sized mills.

Given the relatively small size of the mill by open pit mining standards, there is an opportunity to investigate modular milling options, which would potentially reduce the mill capital cost required, and it is recommended to investigate these options in detail in higher levels of study.

Table 21-2 summarizes the total evaluated processing capital costs for the Elk Gold 900 t/d processing facility of $23.8 million.

Table 21-2: Elk Gold Process Plant Capital Cost Estimate (900 t/d)

Item	Cost ($ million)
Direct Costs	11.6
Field/Construction Costs	4.1
EPCM	5.0
Working Capital	1.5
Contingency @ 25% of the Total Constructed Cost	1.6
Total	**23.8**

Notes: Cost estimates are considered accurate to ±30% for this PEA.

The estimated direct costs are $11.6 million while the indirect costs are estimated to be $12.2 million. No major plant rebuild or expansion are considered in the evaluated production scenario due to the short production life of 10 years. An allowance of $200,000/a of sustaining capital has been included. No allowance for salvage value was made.

21.1.4 Tailings Management

The PEA plan envisages that tailings material from the 900 t/d mill will be dewatered and comingled with non-mineralized rock in the RMSF; therefore, this estimate does not include a tailings management facility.

21.1.5 Infrastructure Capital Cost

The capital cost of the limited amount of infrastructure required to initiate mining operations is captured in the mine capital cost estimate, and no additional infrastructure cost is allocated here.

21.1.6 Surface Water Control

The capital cost of surface-water collection ditches and water-management structures are captured in Section 21.1.2 of this report, and no additional capital cost is allocated here.

21.1.7 Reclamation and Closure

Reclamation and closure costs are included at the end of the mine life and are assumed to be $10 million, which includes the re-sloping of mine facilities, capping facilities with till and topsoil, and revegetation. It also includes removal of mine facilities and preparation of the Project site for long-term closure.

21.2 Operating Cost Estimate

21.2.1 Mine Operating Cost

The open pit mining activities for the Elk Gold Project were assumed to be undertaken by an owner-operator rental-lease fleet as the basis for the PEA. The cost estimate was built from first principles as well as the author's experience with similar-sized open pit operations. Equipment efficiency was estimated based on on-site conditions (e.g., estimated haul routes for each phase). Local labour rates (for operating, maintenance, and supervision/technical personnel) and estimates on diesel fuel pricing ($0.80/L) were used for the mining cost estimate.

The unit cost estimate for this Project, as shown in Table 21-3 is $3.03/t mined.

Table 21-3: Mine Operating Cost Estimate

Open Pit Mine Operating Unit Cost	LOM Cost ($ million)	Unit Cost ($/t mined)	Unit Cost ($/t processed)
Drilling	19.3	0.24	8.5
Blasting	38.3	0.46	16.4
Loading	43.7	0.52	18.7
Hauling	108.4	1.28	46.1
Roads/Dumps/Support Equipment	22.2	0.26	9.3
General Mine/Maintenance	3.5	0.04	1.5
Supervision & Technical	19.9	0.23	8.3
Total Open Pit Operating Cost	**255.4**	**3.03**	**108.6**

The mining cost estimates encompass open pit and dump operations, road maintenance, mine supervision, and technical services.

21.2.2 Processing Operating Cost

Operating costs (including tailings handling) were estimated $21.03/t of processed material based on process flowsheet for the Elk Gold deposit at 900 t/d plant capacity. This includes labour, consumables, supplies, and contingency.

Table 21-4 summarizes the breakdown of operating expenses.

Table 21-4: Elk Gold Process Plant Operating Cost Estimate (900 t/d)

Expense Item	$/t of Process Feed
Labour	9.18
Power	5.22
Consumables & Supplies	1.78
Contingency (30%)	4.85
Total	**21.03**

21.2.3 Plant Feed Highway Haulage

The PEA Mine Plan envisages hauling the process plant feed to an off-site toll mill for the first three years of operations. This highway haulage will be carried out by contractor B-train style dump trucks; the operating cost of this activity is estimated to be $0.11/t/km over 67 km which equates to $7.37/t processed.

21.2.4 General and Administrative

This PEA cost estimate for general and administrative (G&A) costs includes the salary and burden of an administration staff, costs associated with mine rescue, and general supplies. Table 21-5 shows a summary of the G&A costs.

Table 21-5: Summary of G&A Costs

G&A	Annual Cost ($ '000s)	Unit Cost ($/t processed)
Staff	846	2.61
Mine Rescue	240	0.74
Supplies	150	0.46
Contingency (30%)	371	1.15
Total	**1,608**	**4.96**

22 ECONOMIC ANALYSIS

22.1 Cautionary Statements

22.1.1 Certainty of Preliminary Economic Assessment

This PEA is preliminary in nature. It 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, and there is no certainty that the PEA will be realized.

22.1.2 Mineral Resources are Not Reserves

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

22.1.3 Note on Inferred Resources

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

Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

22.2 Introduction

This PEA-level technical economic model was developed for the Elk Gold Project and it contemplates mining and processing material at 900 t/d at an average head grade of 5.3 g/t Au and 8.7 g/t Ag. Summary parameters and economic results are presented in Table 22-1.

Table 22-1: Summary of Economic Parameters and Results

Parameter	Unit	Value
Mill Feed	t ('000s)	2,397
Payable Au Produced	oz ('000s)	373
Payable Ag Produced	oz ('000s)	609
Long-term Au Price	US$/oz	1,600
Long-term Ag Price	US$/oz	18.11
Gross Revenue	$ M	788
Treatment and Refining Costs	$ M	2.5
Royalty	$ M	16.1
Operating Costs	$ M	323
Operating Surplus	$ M	464
LOM Capital Costs	$ M	60.3
Economic Results		
Pre-tax $NPV_{5\%}$	$ M	318
After-tax $NPV_{5\%}$	$ M	190.8
Non-discounted Payback from Production Date	months	6

Notes: All figures are expressed in Canadian dollars unless otherwise stated.

22.3 Key Assumptions

The following production-related assumptions have been applied to the technical economic model:

- Production rate at maximum of 900 t/d, 365 d/a
- Pre-production period of two months is included in Year 1 of the model.

In addition, the following general assumptions have been applied for mine design and economic evaluation:

- A base case discount factor of 5% has been applied for NPV calculations
- Metal prices over years one to five are modelled as described in Section 19
- A long-term sales price of $1,600/oz Au
- A long-term sales price of $18.11/oz Ag
- Estimates of royalties payable as described in Section 22.3.5.

22.3.1 Construction Schedule

For the purposes of economic evaluation, no pre-production period is modelled on an annual basis, and all production costs are incurred in Year 1.

22.3.2 Mine Life

The LOM is 10 years, with the first three years toll-milling at 70,000 t/a followed by seven years of milling on site at 324,000 t/a. Total material mining rates increase as the processing rate increase and as the pit phases increase in strip ratio later in the mine life.

22.3.3 Commodity Pricing

The base case economic evaluation uses long-term commodity prices of $1,600/oz for gold and $18.11/oz for silver. Sales prices have been applied to all LOM production without escalation or hedging. The detailed annual price forecasts applied in the economic evaluation are summarized in Section 19.

22.3.4 Revenue Calculations

Revenue is determined by applying selected metal prices to the annual payable metal contained in concentrate, minus payable percentages, transportation and refining charges, and royalty payments.

22.3.5 Royalty Calculations

There is a 2% NSR royalty payable to Almaden. The NSR value was calculated as a percentage of the gross metal revenues net of off-property costs, transportation, and refining charges.

Table 22-2: Base Case LOM Revenue Parameters

Parameter	Unit	LOM
Milled Feed	t ('000s)	2,397
Contained Au	oz Au ('000s)	406
Contained Ag	Ag ('000s)	677
Metallurgical Recovery		
Au	%	92
Ag	%	90
Recovered Metals		
Au	oz ('000s)	373
Ag	oz ('000s)	609
Refinery Payable		
Au	%	97
Ag	%	90
Payable Metals in Concentrate		
Au	oz ('000s)	362
Ag	oz ('000s)	548
Revenues		
Commodity Sales Prices		
Au	US$/oz	1,600
Ag	US$/oz	18.11

Parameter	Unit	LOM
Value of Metal in Concentrate Before Deductions		
Au	$ M	817
Ag	$ M	15.0
Gross Revenue from Concentrate Before Deductions	$ M	832
Revenue from Concentrate after Payable Deductions		
Au	$ M	792
Ag	$ M	13.5
Revenue from Concentrate after Payable Deductions	$ M	806
Treatment and Refining Costs		
Au Refining Costs	*$/oz*	*6.00*
Ag Refining Costs	*$/oz*	*0.50*
Treatment and Refining Costs	$ M	2.5
NSR Royalty	$ M	16.0
Net Revenue from Concentrate after All Deductions	$ M	788

Notes: All figures are expressed in Canadian dollars unless otherwise stated.

22.4 Taxation

The Project was evaluated on an after-tax basis using a simple tax model appropriate for a PEA-level evaluation, and estimating taxes payable in the Province of British Columbia and Federal tax. Tax depreciation schedules were simplified for the purposes of analysis and do not precisely reflect the expected detailed tax depreciation for the Project. Total tax payments are estimated to be $161 million over the LOM.

22.5 Base Case Valuations

The primary economic evaluation measures used were total LOM cash flow, NPV of this cash flow at a 5% discount rate ($NPV_{5\%}$), and the payback period on a non-discounted basis. Table 22-3 summarizes the high-level economic outputs from the modelling. Note that payback is quoted from the commencement of production.

A summary of annual cash flows produced from the technical economic model at a gold price of $1,600/oz and a silver price of $18.11/oz are presented in Table 22-3.

Table 22-3: Annual Cash Flow Summary

	Unit	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Open Pit Production												
Resources Mined	dmt ('000s)	2,397	70	70	70	324	324	324	324	324	324	243
Non-Resources Mined	dmt ('000s)	83,617	2,056	1,940	2,358	3,003	4,041	13,749	14,126	1,7075	16,492	8,777
Au	(g/t)	5.26	11.4	7.2	7.4	5.5	5.0	5.2	5.6	5.1	3.8	4.0
Ag	(g/t)	8.78	10.5	6.1	7.0	6.3	7.6	9.2	11.6	10.9	7.9	8.4
S	(%)	0.71	0.8	0.1	0.1	0.7	0.5	0.7	1.0	0.8	0.8	0.8
Strip Ratio	(nr:r)	34.9	29.4	27.7	33.7	9.3	12.5	42.4	43.6	52.7	50.9	36.1
Au Mined	oz ('000s)	406	25.6	16.3	16.6	57.0	51.9	54.7	58.5	53.3	40.1	31.6
Au Recovered	oz ('000s)	373	23.5	15.0	15.3	52.4	47.7	50.3	53.8	49.1	36.9	29.1
Ag Mined	oz ('000s)	677	23.5	13.8	15.9	65.9	78.9	96.0	120.7	113.9	81.9	66.0
Ag Recovered	oz ('000s)	609	21.2	12.4	14.3	59.3	71.0	86.4	108.6	102.5	73.7	59.4
Revenue												
Au Price	US$/oz		1,777	1,717	1,645	1,625	1,600	1,600	1,600	1,600	1,600	1,600
Au Price	$/oz		2,401	2,320	2,223	2,196	2,162	2,162	2,162	2,162	2,162	2,162
Au Payable	$ '000s	817,342	56,481	34,751	33,997	115,136	103,178	108,717	116,371	106,063	79,776	62,873
Ag Price	US$/oz		19.27	19.07	18.52	18.27	18.11	18.11	18.11	18.11	18.11	18.11
Ag Price	$/oz		26	26	25	25	24	24	24	24	24	24
Ag Credit	$ '000	14,971	552	320	357	1,465	1,739	2,114	2,658	2,510	1,804	1,453
Cost of Gold Revenue												
Gold Payable Deduction	$ '000	24,520	1,694	1,043	1,020	3,454	3,095	3,261	3,491	3,182	2,393	1,886
NSR	$ '000	15,812	1,093	672	658	2,227	1,996	2,103	2,251	2,052	1,543	1,216
TC/RC	$ '000	2,238	141	90	92	315	286	302	323	294	221	174
Total Cost of Revenue	**$ '000**	**42,570**	**2,928**	**1,805**	**1,769**	**5,996**	**5,378**	**5,666**	**6,065**	**5,528**	**4,158**	**3,277**
Cost of Silver Revenue												
Silver Payable Deduction	$ '000	1,497	55	32	36	146	174	211	266	251	180	145
NSR	$ '000	263	10	6	6	26	31	37	47	44	32	26
TC/RC	$ '000	304	11	6	7	30	36	43	54	51	37	30
Total Cost of Revenue	**$ '000**	**2,065**	**75**	**44**	**49**	**202**	**240**	**292**	**367**	**346**	**249**	**201**
Gross Revenue Gold	$ '000	774,771	53,553	32,946	32,227	109,140	97,801	103,050	110,305	100,535	75,618	59,596
Gross Revenue Silver	$ '000	12,906	476	276	308	1,263	1,499	1,822	2,291	2,163	1,555	1,253
Capital Costs												
Total Owner's Costs	**$ '000**	**16,165**	**4,515**	**4,600**	**4,390**	**380**	**380**	**380**	**380**	**380**	**380**	**380**
Total Capital Cost	**$ '000**	**44,094**	**6,945**	**0**	**26,049**	**200**	**200**	**200**	**200**	**150**	**5,100**	**5,050**
Operating Costs												
Mining Cost (Total Material)	$ '000	255,395	3,401	8,255	9,640	11,601	14,296	39,840	41,239	48,239	49,841	29,043
Highway Haulage	$ '000	1,548	516	516	516	0	0	0	0	0	0	0
Mineral Processing	$ '000	56,867	0	0	0	8,424	8,424	8,424	8,424	8,424	8,424	6,323
Toll Milling	$ '000	9,450	3,150	3,150	3,150	0	0	0	0	0	0	0
Total Operating Cost	**$ '000**	**323,259**	**7,067**	**11,921**	**13,306**	**20,025**	**22,720**	**48,264**	**49,663**	**56,663**	**58,265**	**35,365**
Total Cash Costs, Net PBC	**$ '000**	**370,612**	**18,051**	**16,245**	**43,437**	**19,342**	**21,801**	**47,022**	**47,952**	**55,030**	**62,190**	**39,543**
	AISC US$/oz	735										
Net Before Tax Cash Flows	$ '000	404,160	35,501	16,701	(11,210)	89,798	76,000	56,029	62,354	45,505	13,428	20,053
Tax Expense	$ '000	161,641	13,316	6,836	382	32,034	30,092	22,288	24,957	18,263	5,415	8,057
After Tax Cash Flows	$ '000	242,519	22,185	9,865	(11,592)	57,764	45,908	33,741	37,396	27,242	8,013	11,997

Notes: The discount rate used is 5%, all numbers are in real C$. All figures are expressed in Canadian dollars unless otherwise stated.

22.6 Certainty of Preliminary Economic Assessment

The PEA is preliminary in nature. It 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, and there is no certainty that the PEA will be realized.

22.7 Sensitivities

A sensitivity analysis was performed on the base-case settings by applying sensitivity to changes in commodity prices and total cash costs. The results of this analysis are presented in Table 22-4.

An optimized mining and processing plan was not developed for each case.

Table 22-4: Effect of Variation in Revenue and Capital and Operating Costs on After-Tax NPV5%

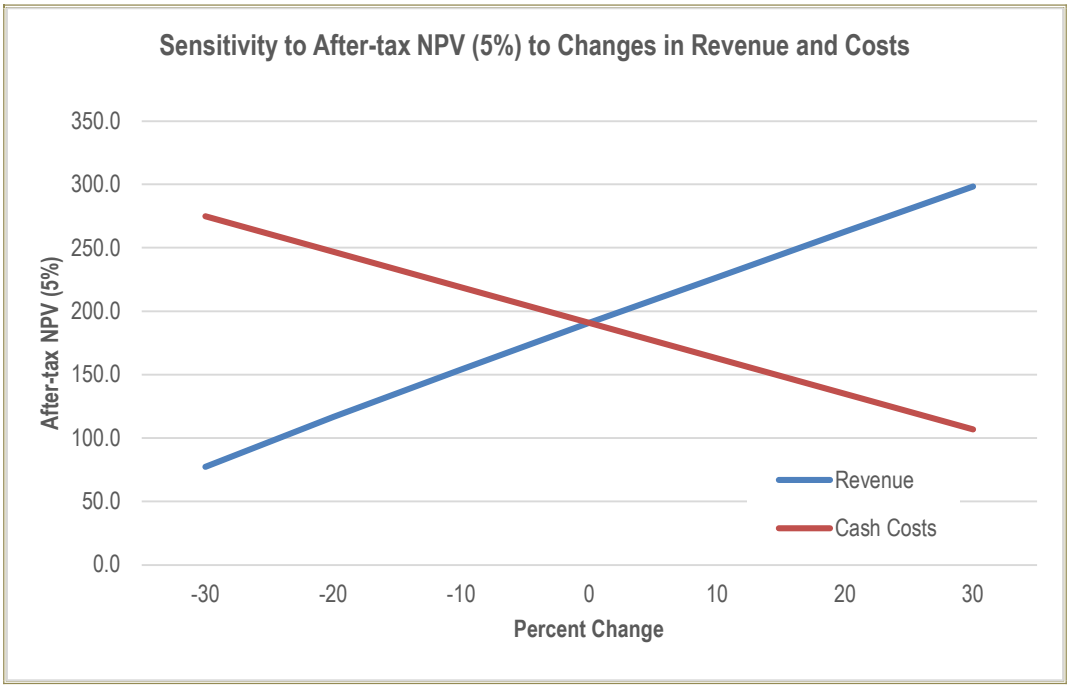
		Total Project Operating and Capital Costs (%Change)						
		30%	**20%**	**10%**	**0%**	**−10%**	**−20%**	**−30%**
	−30%	−6.5	21.5	49.5	77.5	105.5	133.5	161.5
LOM Revenue (%Change)	**−20%**	32.6	60.6	88.6	116.6	144.6	172.6	200.6
	−10%	70.1	98.1	126.1	154.1	182.1	210.1	238.1
	0%	106.9	134.9	162.9	**190.9**	218.9	246.9	274.9
	10%	142.8	142.8	198.8	226.8	254.8	282.8	310.8
	20%	178.7	206.7	234.6	262.6	290.6	318.6	346.6
	30%	214.5	242.5	270.5	298.5	326.5	354.5	382.5
	30%	214.5	242.5	270.5	298.5	326.5	354.5	382.5

Sensitivity analysis indicates that to achieve a breakeven after-tax NPV the price of gold and silver would have to drop by approximately 49% to $784/oz Au and $9/oz Ag (Table 22-5). Conversely, total cash costs would have to increase 68% to achieve a break-even after-tax NPV.

A graphical illustration of sensitivity analysis is presented in Figure 22-1.

Table 22-5: Effect of Variation in Gold Price on Project NPV

Long-term Gold Price (US$/oz)	Pre-tax NPV (5%) (M$)	After-tax NPV (5%) (M$)
2,200	543.2	325.2
2,000	468.2	280.4
1,800	393.3	235.7
1,600	**318.3**	**190.9**
1,400	243.3	144.9
1,200	168.2	97.1



Source: AKF Mining Services, August 2020

Figure 22-1: Sensitivity Analysis

22.8 Conclusions and Recommendations

The economic analysis of the PEA indicates that a conventional open pit mining and milling operation is potentially economic, with a base-case after-tax $NPV_{5\%}$ of $190.8 million.

23 ADJACENT PROPERTIES

There are several mineral explorations claims in the vicinity of the Property; however, none are considered significant.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Bulk Sample Mining—Process

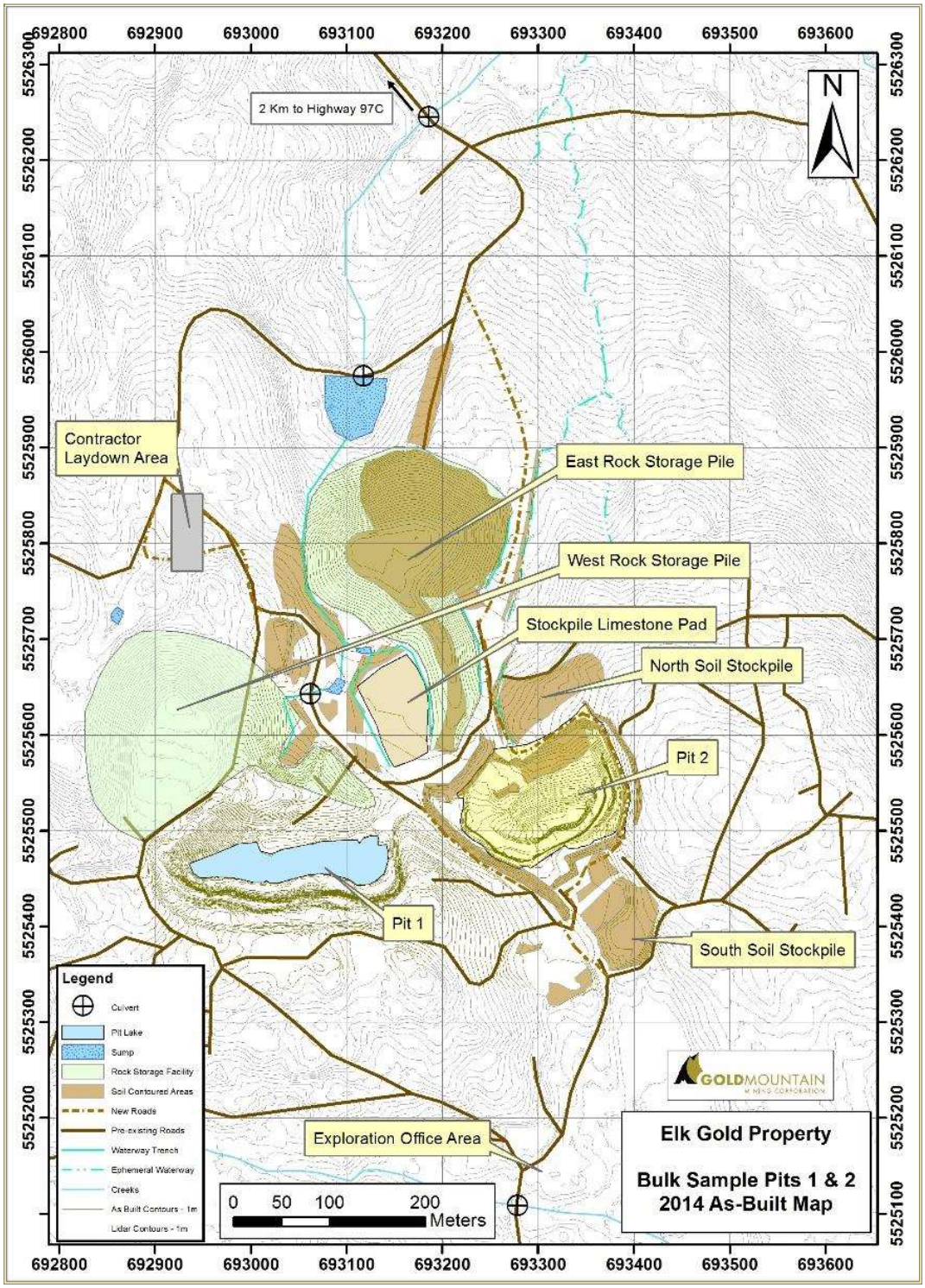
An open pit test mining operation initiated in 2012 was permitted under *Mines Act* Permit M-199 to collect a 4,000 m^3 bulk sample. The work was completed to confirm results of the preliminary economic assessment as outlined in the 2011 Technical Report (Pooley et al., 2011). The test mining and bulk sample program will be used to refine the mining technique and confirm parameters for possible full-scale operations.

Mining that started in 2012 was suspended during 2013 while a 500-tonne test sample was processed. Mining restarted in 2014 and the approved mine plan was completed (Figure 24-1), though additional room remains within the allowable 4,000 m^3 bulk sample allotment.

The bench that was being constructed was divided into two separate sections, the production area and the wedge area (Figure 24-2). Four-inch blast holes in the production area were drilled on a 10 ft x 10 ft offset pattern to the full depth of the subsequent bench with no sub-blast. Pre-packaged emulsion sticks with primers were used a blasting agent and approximately 2 m was left uncharged at the top of the hole. Stemming which consisted of ½" crushed rock, completed the hole. An average powder factor of 0.7 kg/bcm was used on both production and wedge blasts.

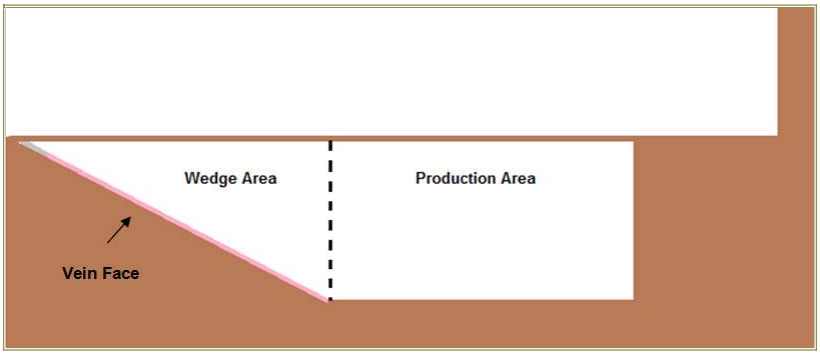
The blasted material in the production area was excavated and sent to the rock storage facility. By excavating this non-mineralized rock material, a relief area was established for blasted wedge rock.

Drill holes in the wedge area were drilled to varying depths depending on their location relative to the vein face. The depth of these drill holes was based on the working 3-D Surpac vein model and were adjusted to terminate 1 m above the expected vein depth. Drill hole locations and depth control was by survey using an RTK GPS unit. Drill cuttings in the wedge area were constantly supervised to ensure no vein material was being drilled and blasted. The wedge blast shock-wave fractured but did not disrupt the vein.



Source: Bayshore, August 2020

Figure 24-1: 2014 As-Built Plan Map



Source: Bayshore, August 2020

Figure 24-2: Bench Cross Schematic Section

Vein mining occurred in segments 2.5 m by 10 m in area. Segments, referred to as plates were given unique number names and their perimeters were surveyed by RTK GPS which allowed a 3-D model of the vein to be constructed. The vein thickness was calculated by taking the total tonnes of vein mined as measured by shipped weights, divided by an average of the measured and calculated vein specific gravity to determine the cubic metres mined, which was then divided by the total area of mined plates to arrive at a 0.55 m average thickness. All plates were stitched together in 3-D using Surpac software to produce a mined area solid model as well as create the Pit 2 footwall as-built plans.

Surveying during the 2014 season was carried out by a qualified GMMC employee using a Hemisphere S320 RTK GPS. Accuracy errors during surveying were mitigated automatically through tracking of the satellite constellations periodically from the handheld data collector. A mask angle (cut-off angle) of 13° was implemented to block signals from satellites near the horizon that have the longest run through the atmosphere therefore increasing the error potential. Surveys were also not carried out when the positional dilution of precision (PDOP) measured above 2. Ionospheric and tropospheric errors were negligible due to the proximity and identical weather conditions between the base station and rover during surveys, which also reduced the distance-correlated error factor to a negligible state. Multipath errors were minimized by positioning the base station high up on the hill where it would remain in line of site most of the survey time. Control points were established using existing survey stations as well as adding new ones during the year. Daily checks to ensure equipment accuracy and precision were completed by using the rover to survey known stations around the mine site. This check eliminated the possibility of systematic errors such as incorrect antenna and instrument heights.

Visual grade control was used during the vein removal process to minimize dilution and vein loss. During bulk sample mining a major goal was to strike a balance between grade (dilution and vein recovery) and efficiency. In 2014, it was decided to take more footwall dilution with the vein to increase minor efficiency, and this strategy was successful, keeping grades at a reasonable level. A thin footwall alteration envelope adjacent to the competent footwall rock allowed vein mining simply to proceed until the excavator could no longer scrape material from the footwall face. The mined vein material was trucked to a storage pad, dumped as individual piles, and marked by stakes bearing the plate number. The plate

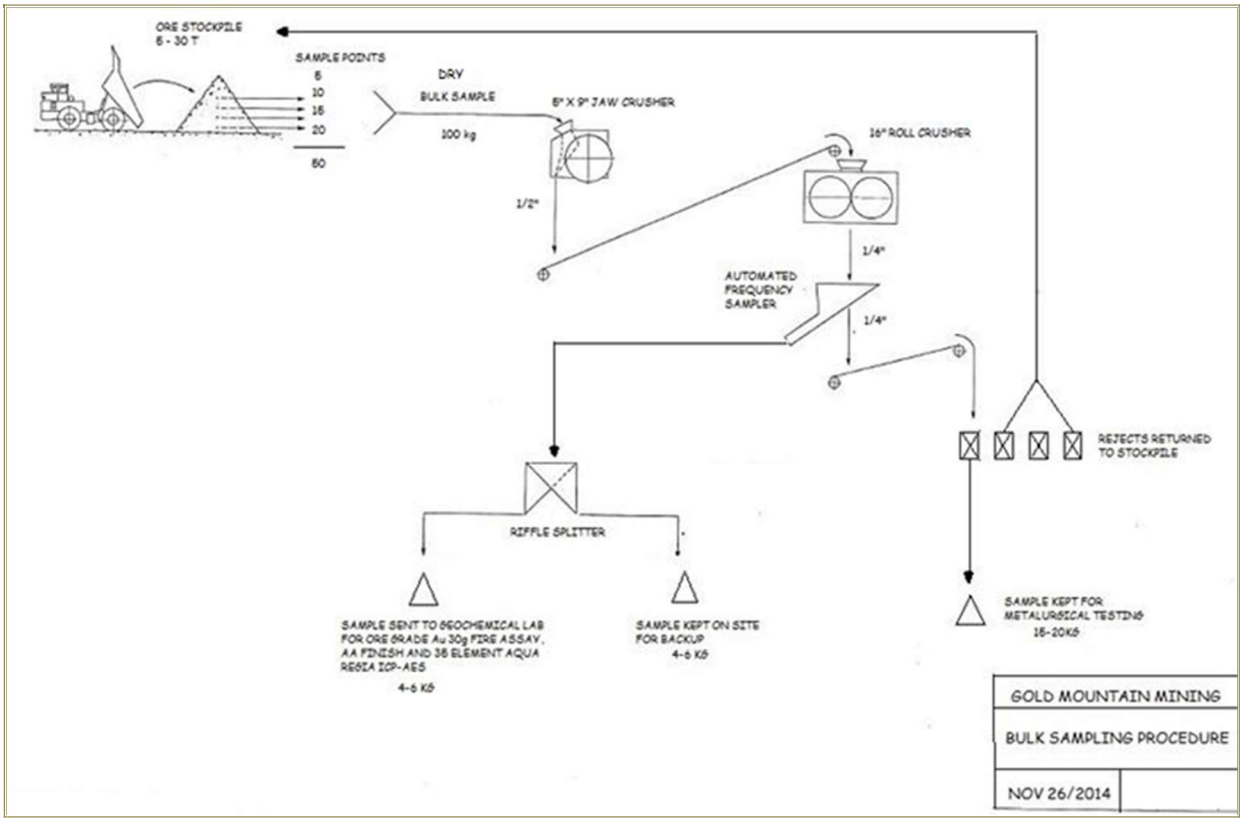
piles are referred to as stockpiles. The stockpile storage pad consists of 3/4" crushed limestone spread 8" thick, over an area of 4,800 m^2 and was installed in 2012 to neutralize any low pH fluids emanating from stockpiles.

Vein material from the bulk sample pit was strategically placed on the pad based on visual grade estimates of the material. Material determined to have the highest grades was grouped together and placed preferentially on top of an area that had been used previously to reduce the loss of high-grade material within the crushed limestone. Transport trucks were loaded on the pad so placement for accessibility to the stockpile was a major consideration.

All stockpiles were surveyed when the corresponding plate mining was completed. Waypoints were entered every meter, or at any major inflection point on the piles' base perimeter and smaller rings spaced at 1 m increments to the top of the pile. A "solids" model of the pile was created in Surpac to establish a volume of material contained. By attributing a specific gravity, using a swell factor to the volume, a wet tonnage was obtained for the entire pile.

Mine progress was documented on a regular basis by surveying and creating as-built plans. Every 2-m bench was surveyed either once completed or during excavation. As-built plans were constructed by taking waypoints around the perimeter of new workings, along either new roads or benches that were established. Distance between survey points depended on topography and alignment but were usually no more than 5 m apart.

Mineralized stockpiles were split into two categories: above and below 15 tonnes. Stockpile names are equivalent to plate names and were derived from bench elevation, vein name, and plate number (e.g., 1608-1300-04). In both categories the material was divided into four horizontal sections to be sampled. All stockpiles were sampled from bottom to top to reduce contamination with the lower section. For stockpiles over 15 tonnes, a spaded shovel was used to sample 20 evenly spaced locations around the bottom section, 15 samples from the section above, 10 samples for the third section and 5 at the top for a total of 50 samples (Figure 24-3). For stockpiles less than 15 tonnes the same procedure was applied at a less frequent interval of 12, 9, 6, and 3 for a total of 30 samples. Material was shoveled into labelled 5-gallon buckets and taken to the drying station. The sampling produced a representative sample weighing between 100 kg and 150 kg.



Source: GMMC, 2015

Figure 24-3: Stockpile Sampling Flow Diagram

At the drying station, custom-fabricated aluminum lined shelves held the sampled material. An industrial fan and de-humidifier were set up in the heated facility to accelerate the drying process. Depending on make-up and amount of material, drying times ranged from 8 to 24 hours. After samples had been dried, they were returned to their respective buckets and the shelves thoroughly cleaned to avoid contamination. Dry samples were stored until approximately eight stockpile samples were ready to be crushed.

A crushing plant consisting of a 4" jaw crusher in tandem with a 4" roller crusher reduced the sample rock to a -1/8" size. An automated frequency sampler split out a target weight of 8 kg of sample material. This material was further split into two 4 kg sub-samples using a Jones Riffle Splitter. One sub-sample was retained at the site as a legacy sample and the other was sent to the lab for analysis.

Samples were sealed and kept in secure storage until shipped to the laboratory by courier or staff employee. Samples sent to the lab were assayed for gold and silver plus 35 element ICP-AES analysis. It was shown with fairly good confidence that bismuth can be used during mining as a double check for receiving gold assays (Benoit et al., 2014), Figure 24-4. Higher bismuth grades correlated with higher gold, so if a sample returned low gold and high bismuth results then a re-assay would be warranted. However,

a correlation study of Bi-Au in core samples elsewhere on the Property had a very poor resulting correlation indicating that this has limited application for regional property exploration.

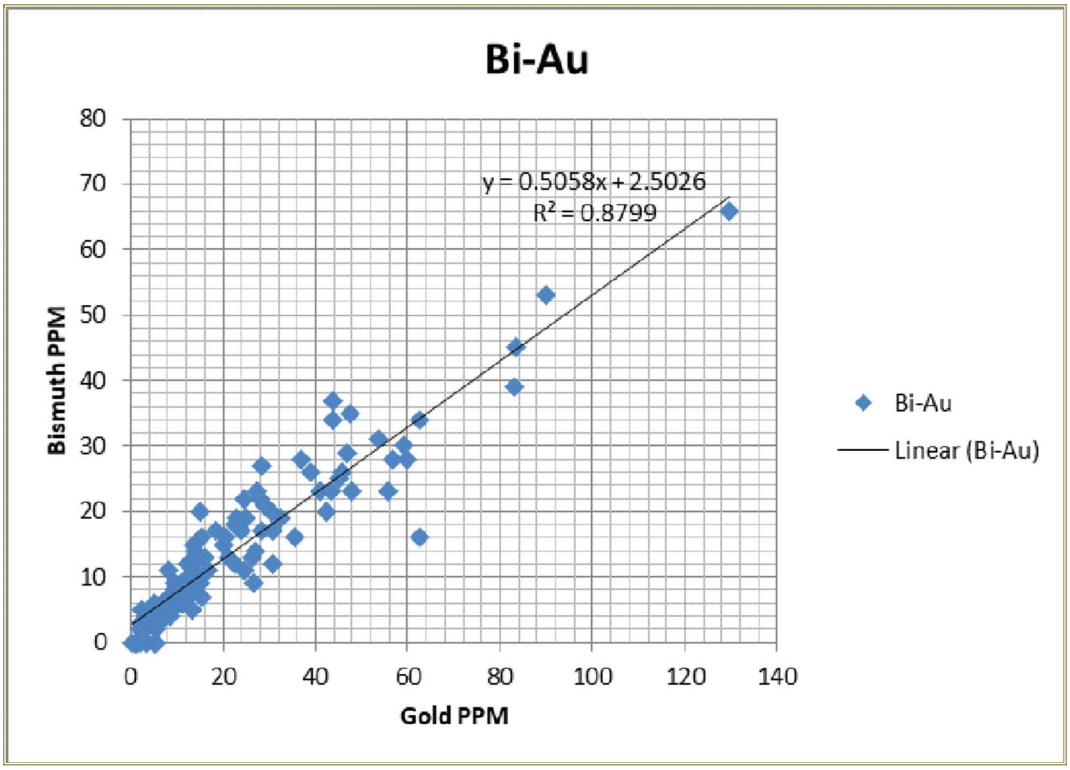


Figure 24-4: Bismuth (ppm) vs. Gold (ppm) in 2014 Pile Samples

Upon receipt of the assay results the entire pile was assigned the grade of the representative sample. Except for one 18-L bucket of raw crushed material, all remaining crushed sample material was returned to its respective stockpile. The retained bucket was sealed, labelled with the pile number, and stored for future testing.

Vein material was shipped to Kinross' Kettle River mill near Republic, Washington. Kinross custom mills "ore" from third parties to supplement feedstock from their Buckhorn Mine. Stockpiled mineralized vein material from the Property was transported twice a day, morning, and evening, using two truck-and-trailer units capable of hauling approximately 30 tonnes each in compliance with U.S. Department of Transportation laws. Truck loading was by a loader equipped with an on-board digital bucket scale that was calibrated daily. Target net weight was a daily total of 120 wet tonnes.

Bulk sample shipments consisted of 1 lot in 2013 and 4 lots in 2014. Each lot consisted of between 600 and 2,500 dry tonnes with each Lot having a specific target grade. A maximum of four stockpiles were mixed together to achieve the target grade for each waiting truck. The stockpile mixing recipe was determined using stockpile tonnages and gold grades to achieve a tonnage weighted shipping grade. The recipe was recorded on paper for each shipment and a tracking spreadsheet updated daily to maintain a shipment

tracking log. Table 24-1 summarizes the weights and grades of mineralized material shipped during 2013 and 2014.

Table 24-1: Shipped Vein Material in 2013 and 2014

Year Mined/Shipped	Tonnage		Volume Shipped (m^3 with swell)	Volume Mined (banked m^3)	Average Gold Grade		Total Gold Contained	
	(dry tons)	(dry tonnes)			(g/t)	(troy oz/ton)	(g)	(troy oz)
2014/2014	6,715.8	6,092.5	2,933.4	2,256.5	17.03	0.497	103,782	3,337
2012/2014	95.7	86.8	41.8	32.2	3.46	0.101	301	10
2012/2013	460.0	417.3	200.9	154.6	13.75	0.401	5,736	184
Total Shipped from Pit 2	**7,271.6**	**6,596.7**	**3,176.2**	**2,443.2**	**16.65**	**0.486**	**109,819**	**3,531**
1990s/2013	39.1	35.4	17.1	0.0	14.66	0.428	520	17
1990s/2014	1,245.9	1,130.2	544.2	0.0	4.09	0.119	4,624	149
Total 1990s Shipped	**1,284.9**	**1,165.7**	**561.3**	**0.0**	**4.41**	**0.129**	**5,143**	**165**
Total Shipped in 2013	**499.1**	**452.8**	**218.0**	**154.6**	**13.82**	**0.403**	**6,256**	**201**
Total Shipped in 2014	**8,057.4**	**7,309.6**	**3,519.4**	**2,288.6**	**14.87**	**0.434**	**108,706**	**3,495**
Total Shipped in GMMC	**8,556.5**	**7,762.3**	**3,737.4**	**2,443.2**	**14.81**	**0.432**	**114,962**	**3,696**

Once the vein material was delivered to the mill; ownership of the shipped material was transferred to the mill owner. Due to the protracted residence time of feedstock within the mill's cyanide circuit, gold poured into doré bars is from a mixture of sources. Therefore, third-party feedstock suppliers cannot be compensated for the actual contained gold within their shipped material. Instead, the mill attempts to determine and pay on a feedstock head grade.

At the mill, head grades were determined by collecting representative samples from the crushed feedstock conveyor belt and creating pulps for assaying. Kinross and GMMC conducted independent assaying and the results compared and a settlement grade agreed on. Umpire lab assaying was used to resolve results that did not closely agree. Table 24-2 provides a summary of the received lot shipments and settlement grades from 2013 and 2014. It should be noted that with each succeeding lot the agreement between the assayed amounts diverged substantially. Introduced errors could include changing homogenization techniques, both in sample preparation and subsampling in the lab, or differing flux preparations of the assay sample. Despite the assay differences, a head grade was agreed upon either by reliance on Umpire sampling or averaging the grades. All grades noted in the above and below tables are based upon the agreed head settlement grades.

Table 24-2: Lot Shipment Summary

Lot #	Shipment Year	Wet Tonnage (t)	Dry Tonnage (t)	Wet Tonnage (t)	Dry Tonnage (t)	Settlement Grade (g/dry tonnes)	Settlement Grade (troy oz/t)
1	2013	476.5	452.8	525.3	499.1	13.82	0.403
2	2014	1,405.6	1,329.4	1,549.4	1,465.4	14.09	0.411
3	2014	2,693.9	2,568.3	2,969.5	2,831.1	18.99	0.554
4	2014	2,504.8	2,377.7	2,761.1	2,621.0	12.34	0.360
5	2014	1,102.0	1,034.1	1,214.8	1,139.9	11.45	0.334
Total		8,182.7	7,762.3	9,020.0	8,556.5	14.81	0.432

24.2 Bulk Sample Mining—Geology

Visualizing the 3-D nature of a mineralized deposit and calculating a resource that can be relied upon requires an understanding of the factors that affect mineralization. These include geometry, structure, form, and grade distribution as well as the relationship between actual and drilled grades.

To computer model the geology within the bulk sample pit the veins were separated into distinct and separate entities in order to create valid solids. This, however, was not the case observed during mining. Anastomosing veins that pinch, swell, separate, and merge were observed throughout mining. These complexities were expected; however, some features of the deposit were observed that were not modelled or fully understood during planning stages of the bulk sample. For a discussion of these complexities bulk sample Pit 2 can be roughly divided into three sections: east, central, and west.

The central pit is the largest section along strike and in this area target mineralization was expected to occur either as a distinct quartz vein with sulphides, or as a clay-pyrite gouge. Both were present throughout the pit and in this central area the solid vein model was in general agreement. Mineralization here was mainly concentrated in the 1300 vein while the 1302 vein occurred sporadically one to several meters in the hanging-wall and was recovered where possible.

The east pit generally refers to the two eastern-most plates, and within this area gold mineralization was found in both gouge and vein form, although more abundantly in vein/veinlets. The solid model constructed before mining does not match well with the eastern end of the pit in the sense that the vein begins to splay into several smaller veinlets. Additionally, where they could be distinguished the hanging-wall vein (1302) was often the wider and higher grade of the two veins on the east end of the pit. Commonly the mineralization was observed as three or more veinlets anastomosing across 0.5 m to 2 m in width within a silicified quartz monzonite host rock. This anastomosing structure along with the silicification made recovery a difficult task using the Property's normal mining procedures, and additional excavation techniques were needed, such as a single tooth ripper and/or hydraulic hammer. Both techniques were moderately successful at recovering the mineralization but they greatly reduced mining efficiency.

The western pit area includes a strongly altered subvertical mafic dyke that bordered the westernmost plates of the bulk sample pit. This dyke was termed RBF by Fairfield geologists, the "F" standing for "Fault." Drill holes trying to penetrate the RBF often returned highly altered and sheared rock in the core which was interpreted to be fault-gouge along the edges of the dyke.

The area west of the RBF was used as the main ramp access to the pit, and several veins that were encountered while excavating the ramp in 2012 were stockpiled and shipped. However, the plate locations were not surveyed with much accuracy as a full-time surveyor was not present on-site during mining, and often the area had been excavated before surveyors arrived. These veins were approximately 5 cm to 30 cm wide quartz veins with 5% to 25% coarse grained sulphides and their associated stockpiles yielded grades from 0.98 to 8.6 g/t.

The quartz monzonite host rock in the western area had moderate to strong propylitic alteration, increasing toward the dyke, which made the rock softer and easier to excavate. The general trend in this western area was a steepening of the mineralized vein as well as a widening of the mineralized zone. Mineralization often occurred as a wide quartz vein with 10% to 50% sulphide, although this vein often split into two or three splays widening the mineralized envelope to up to 1-m perpendicular to dip. The veins were almost ubiquitously loosely attached to the surrounding host rock, likely an effect of the RBF. Recovery of mineralization right to the footwall was simple, which led to quick and efficient mining.

The western area was also not modelled with great accuracy partially because the difficulty drilling in the vicinity of the RBF limited the number of drilled vein intercepts, and partially because the vein was assumed to keep a fairly consistent dip. Fortunately, the changing dip did not affect vein recovery because the ground was well fractured by blasting and therefore easily excavated. The dyke was not mineralized, which suggests that it was emplaced post-mineralization. No significant lateral vein displacement was noted on either side of the dyke, even though wall rock adjacent to the dyke was highly fractured parallel to the dyke and an argillic alteration zone was observed in the quartz monzonite host rock surrounding the dyke edges.

Shear-related geometry was observed during mining and this "rollover" or "step down" geometry was observed throughout the pit on the scale of 1 m to 10 m, although it was most pronounced in the immediate vicinity of the RBF on the west end of the pit. This geometry, often associated with a Riedel shear system as suggested by Alexander (2010), was repeated in several iterations down dip in the west pit and was associated with the widest mining widths in the whole operation. This could mean that there was plastic-like deformation of the vein around the time of formation. This geometry was also observed during the 1990s mining and has been referred to as a major "rollover" or "hinge" under Pit 1, where the 1300 vein steepens in dip. Given how common this geometry is, the change in dip was anticipated and aided in vein recovery. A small amount of the vein footwall mining was deemed necessary to allow full recovery of these step-downs and the mining width was simply widened to accommodate this need.

It is likely that "rollover" or "hinge" geometry was formed by shearing during formation of the quartz vein, but the mineralized gouges are strong evidence that shearing also occurred after vein formation. These

gouges, which are strongly associated with gold, consist mainly of pyrite, ground quartz, and very fine clay minerals. They are typically dark grey in colour, described as "sooty" and gold grade appears to be correlated with sulphide content. Mineralized gouges occur throughout the pit with as much variety as the vein in terms of width, orientation, and gold grade. They are often unconsolidated, with strong phyllic alteration in the surrounding host rock in almost all cases. Several assay samples taken at the beginning of 2014 bulk sample mining helped confirm that these gouges contain as much gold as the quartz veins.

24.3 Bulk Sample Mining—Resource Modelling

One goal of completing the bulk sample program was to evaluate resource estimation methods and to streamline resource modelling for future mining on the project. The resource methods used included a block model, a polygonal model, results from pile (plate) surveying and sampling, head grades, and tonnages from mine and mill measurements/assays, as well as final settlement data. The resource comparison was completed only on a limited area that was common to all methods examined, with the eastern area of Pit 2 being common to all three.

24.3.1 Drilled Grades vs. Mined Grades

There was a disparity in expected mined plate grades compared to drilling grades as plate grades were in general higher than drilled grades, and there are two plausible reasons for this. The first is that the presence and importance of mineralized gouges were not fully realized when drilling and modelling the deposit. The gouges are more difficult to recover in a diamond drill hole, which would lead to underestimating the width of these structures and grade due to core loss. Beyond the gouges, the second reason is that the core sample size was very small in comparison to the representative samples of plates. This small sample size is problematic when the nugget effect is considered. It is thought that the gold has formed in randomly distributed clusters, or nuggets, rather than being distributed evenly throughout. This means the smaller the sample size the less likely one of these clusters will be encountered leading to an underestimation of the gold grade. Conversely if a drill hole encounters one of these clusters, the gold grade will be over-estimated. For a nuggety property like Elk the solution is to drill many holes and apply statistics to the results. In general, this theory was confirmed when comparing plates to expected grades generated using drill data. For many areas the gold grade was underestimated by drilling while only a few areas had the grade overestimated. This nugget effect, along with even smaller sample sizes for intersections that consisted mainly of mineralized gouge material led, to generally higher actual grades than expected from analyzing drill data.

24.3.2 Bulk Sample Resource Comparisons

Block Model Resource Estimate—In 2012 AKF Mining Services Inc. (AKF) calculated an in-house block model resource estimate for the 1300 vein within the bulk sample area. AFK took GMMC's 3-D vein solids model and applied Giroux Consultant's 5 m E-W, 2.5 m N-S and 5 m vertical block model using various cut-off grades from 0.01 to 2.00 g/t Au to calculate tonnage, grade and contained ounces at each 0.01 g/t cut-

off increment. This resource estimate was only completed over the eastern half of the bulk sample pit because the solids model did not extend westward to the RBF dyke due to limited and confusing drill data.

Polygonal Model Resource Estimate—A simple polygonal model parallel to the vein was created as a check against the block model. There are several iterations for this polygonal model, originally produced by Jim McCormack, P.Eng. for Fairfield. That model covered a potential eastern extension (Pit 2) to the 1990s open pit and underground mining effort (Pit 1). McCormack updated his model for GMMC with new drill data prior to Pit 2 bulk sample mining commencing in 2012. GMMC completed detailed vein remodelling in the Pit 2 area between the end of the 2012 mining and start of 2014 mining. The polygonal model was also updated in-house at the same time with what was believed to be the most accurate historical information. Later iterations accounted for trench sample location and width changes.

Polygons were created surrounding drill intersection pierce points of the 1300 vein on a two-dimensional plane to create Thiessen polygons that have boundaries halfway between the pierce points. Using the planned pit model these polygons were clipped to include only what would be mined. Each polygon was then adjusted to an estimated vein dip rather than determining the dip of each polygon. This dip adjustment gave a sloped surface area for each polygon that could be used in further calculations. Each polygon was assigned a width and grade based on the drill intercept at its centre. The width was multiplied by the sloped surface area to estimate a volume for each polygon. The volume was multiplied by the bulk density to calculate the tonnage, which in turn was multiplied by the grade to determine the estimated contained gold in each polygon. The polygons' tonnages and estimated contained gold were summated to arrive at an overall estimated gold grade to complete the resource estimate.

Plate and Pile Mining Estimate—As bulk sample mining progressed mineralized vein material was removed in plates. As mentioned above, plates are 2.5-m x ~10-m rectangles that were placed as separate stockpiles on the mineralized rock storage pad. The plates and stockpiles were routinely surveyed after each plate had been completely mined. Each pile was then sampled and sent for assay to ALS.

Using the pile survey, a stockpile volume was calculated in Surpac™ 3-D modelling software. This volume was estimated to be 30% more voluminous than the original rock in place (swell factor). Using a 30% swell factor and a specific gravity of 2.7 the tonnages were calculated for each stockpile. Using the assay from ALS as a grade, the tonnage could be applied to calculate the gold contained within each pile. This information was used to determine blending recipes for vein shipments to the mill and gave a preliminary estimate of the shipped material grade-tonnage to be compared to that received by the mill. Additionally, several historical stockpiles that remained from the 1990s mining were surveyed and sampled to enable calculated pile grade—tonnage estimates, and inclusion within the shipping program.

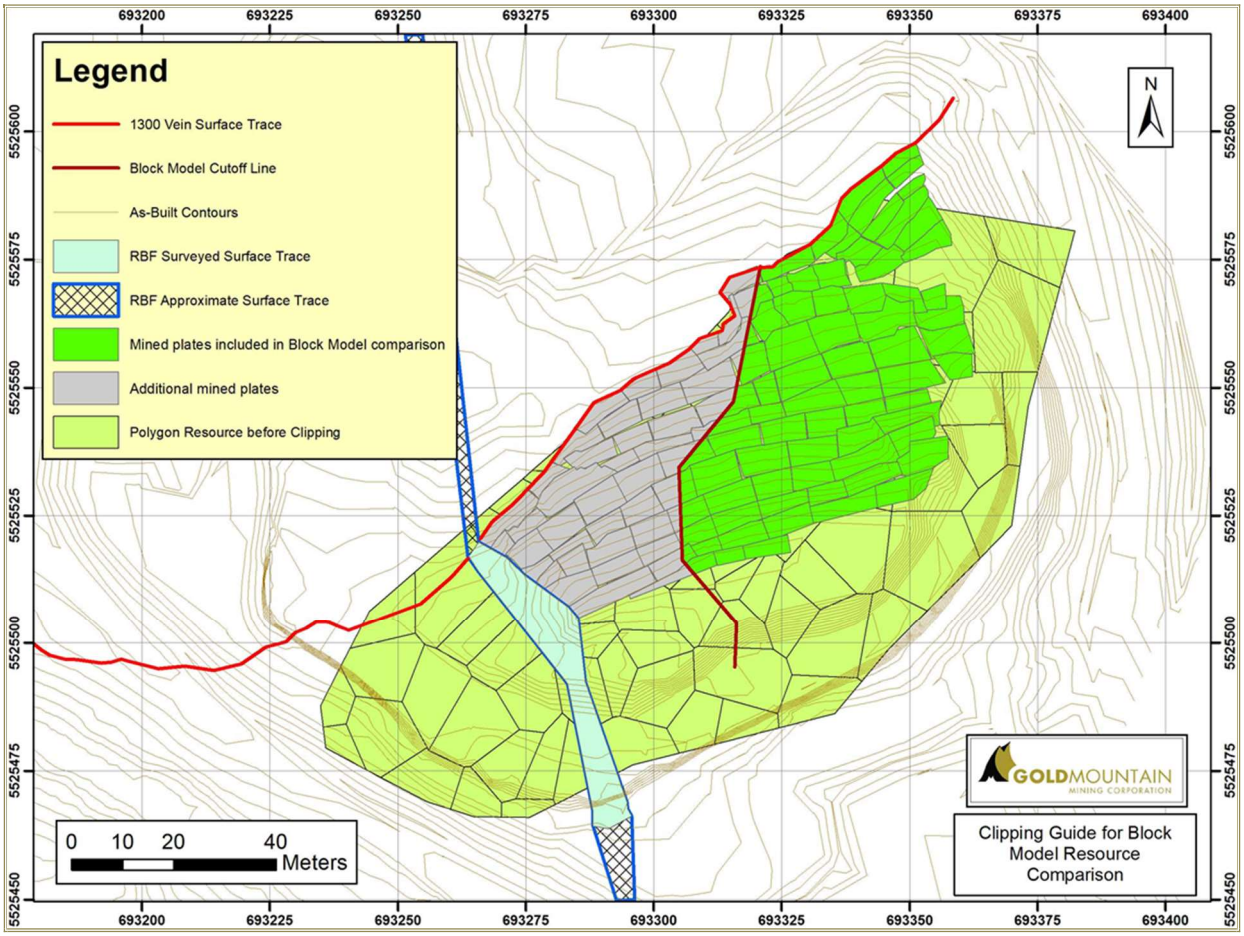
Settlement Mining Results—The final tonnage and grade for each shipped lot used in determining payment is known as the received dry tonnage and settlement head grade. For each of the five Lots shipped during 2013 and 2014 the tonnages and settlement head grades are assumed correct for the purposes of resource comparisons.

Upon truck arrival at the mill the received dry tonnage was calculated using the shipped (wet) vein tonnage determined by truck scale and moisture content determined using a representative sample's as-received and dried weights. A settlement head grade was agreed upon by GMMC and the mill after comparing separate averaged assays for duplicate head grade samples drawn from a single representative sample. These preliminary head grades are known as the mine head grade and mill head grade.

It should be noted that agreement between averaged assays conducted by the two labs became less and less close with each successive lot shipment, from less than 2% difference to over 20% difference. It is unclear as to the cause of this difference, and though the variance between individual assays splits of the same sample were high, the average of those assays should have matched more closely between labs. A variety of methods were employed to reconcile the preliminary head grades and arrive at the settlement head grade including re-homogenizing the representative sample, using Umpire Lab assaying, and averaging the difference between the preliminary head grades. Only re-homogenization produced results satisfactory to both parties.

Resource Estimate Comparisons—To compare the block model resource estimate and/or polygonal resource estimate to the plates and settlement results, the polygons from the resource estimates were clipped in GIS mapping software (ESRI Arcmap) to reflect the same surface area as the mined plates. From that point the plates/piles resource could be compared to block and polygonal resource models by clipping the plate's area by the polygonal and block model outline. The common block model, polygonal model, and mined plates area are shown in Figure 24-5.

Tonnage and grade factors for each lot shipped were applied back onto the piles based on the settlement results. Shipped weights as determined by loader weightometer differed from mill-received weights as determined by truck-scale weights. Similarly, the head grades at the mill differed from pile grades determined at the mine. The lot tonnages determined at the mill were considered correct and a factor calculated to normalize mine and mill values. These factors could then be applied to each plate mined and thus directly compare the settlement results to the block and polygonal resource estimates. It should be noted that in these comparisons the leftover historical piles that were shipped were omitted and therefore the totals do not represent the actual shipped amounts, but only the comparable proportion of the material shipped.



Source: Bayshore, August 2020

Figure 24-5: Clipping Guide for Resource Models Comparison

The three models' tonnages, grades, and contained ounces of gold were then compared and it was found that the block model resource greatly overestimated the actual mining tonnage and significantly underestimated the gold grade. This resulted in an overall underestimation of the total ounces of gold contained in the compared area. This comparison is summarized in Table 24-3 and shown graphically in Figure 24-6, Figure 24-7, and Figure 24-8.

Table 24-3: Block Model, Polygonal Model, Plate and Pile Estimate, and Settlement Results Comparison

Estimate and Results Comparison	Tonnage		Gold Contained		Gold Grace	
	(dry tonnes)	(dry tons)	(grams)	(troy oz)	(g/t)	(troy oz/ton)
Block Model	8,527	9,399	35,667	1,147	4.18	0.122
Polygonal Model	3,401	3,749	51,628	1,660	15.18	0.443
Plates & Pile Grade Estimate	3,148	3,470	50,963	1,638	16.19	0.472
Settlement Results	3,148	3,470	46,089	1,482	14.64	0.427

Notes: Results are to a 2 g/t Au cut-off as that was the minimum grade shipped and also the maximum cut-off calculated in the block model.

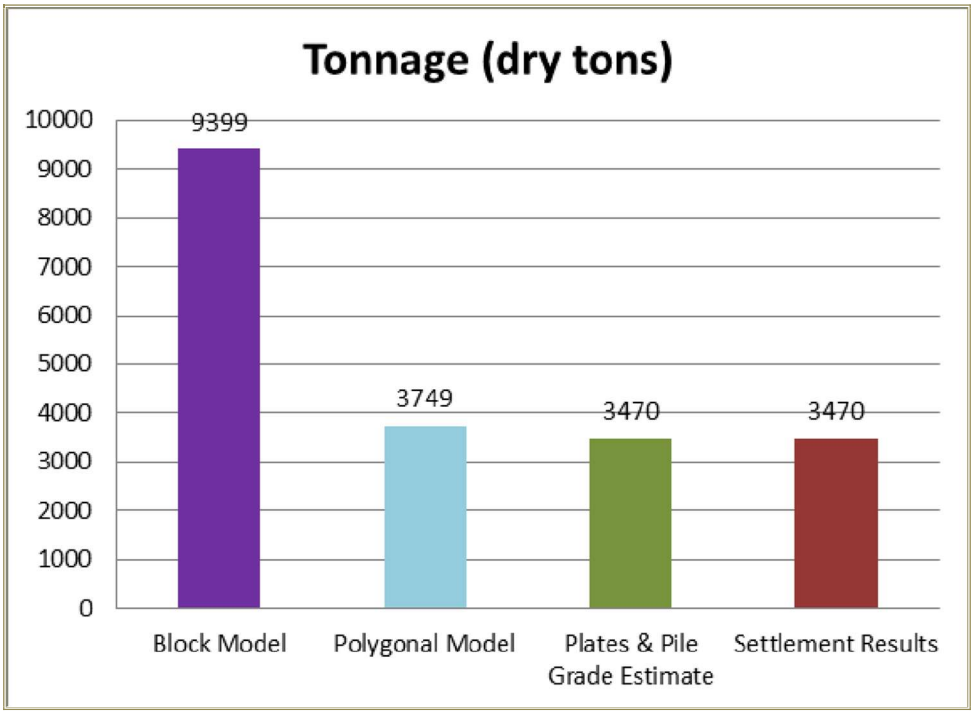


Figure 24-6: Tonnage Comparison—Resource Estimates vs. Actual

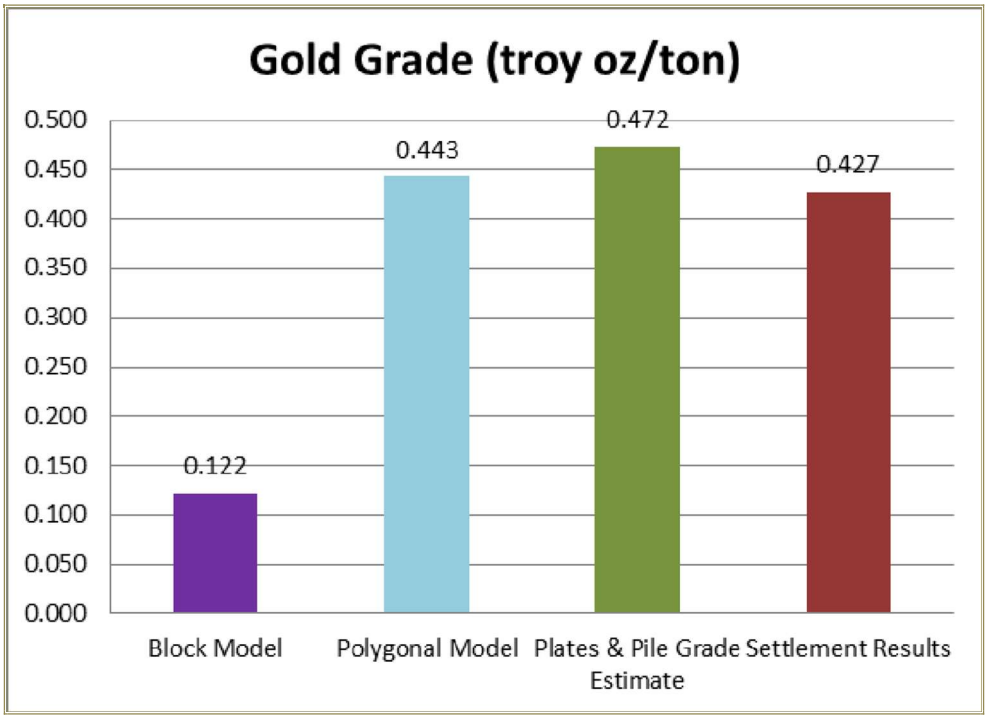


Figure 24-7: Gold Grade Comparison—Resource Estimates vs. Actual

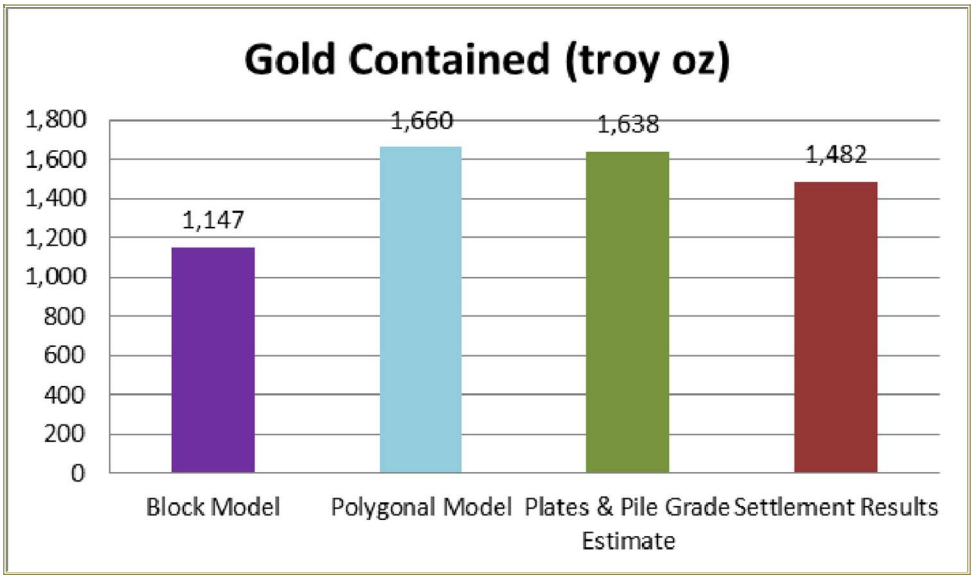


Figure 24-8: Contained Gold Comparison—Resource Estimates vs. Actual

The block model tonnage overestimation is because the solids model was built on the premise that a standard block mining method would be employed and hence a larger volume would be mined. The actual mined amount in this area was 3,470 tons (3,148 tonnes) while the tonnage estimated by the block model was 270% larger at 9,399 tons (8,527 tonnes). Additionally, the mined grade was 0.427 troy oz/st (14.64 g/t) in this area whereas the Block Model grade was estimated at 0.122 troy oz/st (4.18 g/t) or 28.5% of the actual. A focus on lowering the amount of dilution during bulk sample mining meant that this estimate was inaccurate in terms of both tonnage and grade due to overestimation of dilution. However, this does not account for lower total contained gold ounces. The predicted 1,147 troy ounces (35,667 g) were 77% of the actual 1,482 troy ounces (46,089 g) removed in the compared area. This increase in actual ounces realized over what was predicted is similar to that found in 1990s mining in the historical pit (Pit 1).

The polygonal model also overestimated tonnage, but more closely compares to both the plates & piles estimate and actual settlement results and differs significantly from the block model. A polygonal model tends to emphasize grade since the value from a data point (drill intercept grade) applies over the whole of the polygon regardless of the value of the adjacent data point. The polygonal model also included 2012 infill drill results that were missing in the block model and trench results omitted in the block model estimate. Trench results can provide additional data points if the sampling is completed in a rigorous fashion but can also over emphasize grade due to pre-concentration in the surface weathering and erosional environment. It must be stressed that trench results were only included to help guide mining efforts. The polygonal model was 94% of the plates & piles grade estimate and was 112% of the settlement results for contained gold ounces and 108% of the tonnage shipped.

Polygonal Resource vs. Mined Resource—Since the polygonal model more closely reflected actual bulk sample mining results over the restricted block model area it was chosen to compare against the full

mined area results. The polygonal model area still exceeded the mined area so was clipped in GIS mapping software (ESRI Arcmap) to reflect the exact same surface area as the mined plates. The tonnage and grade factors were based on the settlement results for each lot shipped and were applied back onto the piles for each plate mined. This allowed a direct comparison of the settlement results to the polygonal model resource estimate. Again, in these comparisons the shipped historical piles were left out; therefore, the totals do not represent the actual shipped amounts, only the comparable proportion of the material shipped. This comparison is summarized in Table 24-4.

Table 24-4: Polygonal Model, Plate and Pile, and Settlement Comparison Summary

Polygonal Resource Comparison (Includes LG and non-1300 veins)	Volume (m^3)	Tonnage (t)	Tonnage (ton)	Average Grade (g/t)	Average Grade (troy oz/ton)	Gold Contained (g)	Gold Contained (troy oz)
Polygonal Resource	2,657	5,519	6,083	22.68	0.662	125,168	4,024
Pile Sample Grade	3,178	6,597	7,272	18.18	0.530	119,941	3,856
Mine Head Grade	3,178	6,597	7,272	17.17	0.501	113,293	3,642
Mill Head Grade	3,178	6,597	7,272	16.52	0.482	108,985	3,504
Settlement Head Grade	3,178	6,597	7,272	16.65	0.486	109,819	3,531

The results of this comparison show that when a polygonal resource is used there is a tendency to overestimate the gold grade and total contained ounces of gold, while underestimating the tonnage. The total contained gold in the polygonal model was 12.3% higher than that of the settlement results and 4.2% higher than that of the pile sampling estimate completed during mining. The discrepancy between resource estimate and mined numbers seems large at first but when a retroactive grade cap of 100 g/t is applied to the resource estimate the total gold discrepancy shifts from 12.3% to 1.8%. The cap was selected and applied using a multiple trial (back calculation) method and represents the 97th percentile of the 64 polygons used in the study. While only two of the polygons were adjusted in grade, this shifted the results to increase the resource comparison accuracy by over 10%. This analysis highlights a danger of using polygonal models without capping.

While the 100 g/t capping on input grades for the polygonal resource calculation lessens the total gold discrepancy it does not account for the difference in tonnage, which is underestimated in the polygonal resource by approximately 20%. This is likely due to an underestimate in the dilution that took place during the mining process. The widths used in the polygons were on average 0.45 m and the actual widths mined averaged around 0.55 m. If, when calculating the polygonal resource estimate an added mining dilution of 20% at zero grade is applied in addition to a 100 g/t assay cap, then the predicted grade and tonnage would have been within 2% and 1%, respectively of the settlement amounts.

The grade cap on future polygonal resource estimates should be based upon the 97th to 98th percentile of all sample values to ensure that high-grade polygons do not unduly influence calculations. By ranking the influence of each polygon according the total ounces added to the resource, it will become apparent which polygons need further scrutiny.

Adding a mining-width dilution factor will account for the realities of mining this style of vein occurrence using mining method employed during collection of the bulk sample. The footwall material outside of the alteration envelope was hard and could not easily be removed by the excavator, and thus provided a natural lower boundary to the vein. However, this was not the case for the hanging wall, which had both more intense wall rock alteration and blast-induced fracturing. As well, the vein pinched and swelled over short distances, an occurrence not always apparent during mining. Extra hanging-wall material was thus accepted during vein excavation so as not to lose ounces and this action appears to have added 20% more material. This dilution is in addition to that added during drill core sampling.

The drilled vein intersections already had dilution applied during core sampling. A minimum sample width of between 0.3 m and 0.5 m (depending on the year of drilling) oftentimes exceeded the width of the veins, so it included footwall material known from shoulder samples to be virtually devoid of gold. This has implications for future drill programs, where consideration should be given to ending vein samples at the footwall contact and allowing any minimum sample width to range into the hanging wall.

Additional infill drilling, although at greater cost, may improve the input data used by a polygonal estimate. The original polygonal resource estimate prior to 2012 infill drilling was 6,346 oz Au, and this was re-estimated to be 4,545 oz following inclusion of the new assay data over the same area. The implication this has on infill drill targeting is that adjacent high- and low-grade hole pairs should be bisected.

Dilution factors should be further studied during the next round of mining. The following are ideas to be considered:

- Use a simple dilution factor based on current data (20%) and apply it broadly on all samples
- Use a dilution factor but apply it only to the narrower intervals (e.g., everything under 0.5 m receives an additional 20% dilution)
- Set up a number of dilution factors based on the original sample widths (e.g., 0 - 0.3 m = 30%, 0.3–0.4 m = 25%, 0.4–0.5 m = 20%, 0.5–0.55 m = 5%, >0.55 m no factor)
- Core photographs of each polygon drill intercept helped to predetermine a probable mining width and might make the dilution estimate more accurate, rather than applying a generalized dilution factor.

25 INTERPRETATION AND CONCLUSIONS

25.1 Diamond Drilling

Diamond drilling on the Property Siwash North zone in 2010, 2011, and 2012 was designed to:

- Test the continuity of gold mineralized quartz veins along strike and down dip of previous drilling
- Infill existing drill data to allow an upgrade in confidence categories in a resource estimate
- Potentially add gold resources to the known deposit
- Provide vein grade and survey control for bulk sample mining.

Drilling on the Siwash North western extension (formerly known as the Far West zone) helped to better delineate the contacts between the Nicola and Osprey Lake Group rocks which now lies further west than previously thought. Historically it was thought that mineralization was restricted to only the granodiorite and quartz monzonite; the drilling has shown that Nicola volcanic rocks do host gold-mineralized quartz veins that are western extensions of the 1000 series veins. The quartz veins within the Nicola, however, are thin and inconsistent, only occasionally swelling to widths worthy of investigation. Narrower and less continuous veins in the Nicola may be due to the volcanic rocks being less brittle than the intrusive units, and it appears that fractures hosting the gold-mineralized quartz veins did not remain sufficiently open to allow for good vein development. Although mineralization is present in the volcanic rocks, the area that displays the highest grades and largest vein volumes are zones lying east of the Nicola within intrusive host rocks.

Drilling immediately west of Pit 1 has identified additional gold mineralization that may be potentially exploitable. Core samples from drill intersections up to 350 m west of Pit 1 returned assays of >15 g/t Au. Drill spacing would need to be tightened to determine the full potential of this area.

Drill results from the Siwash North south west area confirmed that the 1000 vein series does continue down-dip and does carry gold. The intersections are much lower grade than elsewhere within the zone except for two holes deepest down-dip on the two most southeasterly sections where grades are seen to increase.

The bulk sample area (Pit 2) drilling was relatively shallow and designed to provide grade and survey control for the mining efforts. The veins that this drilling targeted have, for the most part, been removed by the bulk sample mining program.

The 2000 series veins tested at depth along the northeast side of Siwash North zone continue to return gold grades that make this zone an attractive target for future exploration. The veins' steep dip suggests that any mining would be by shallow open pits and underground techniques.

25.2 Bulk Sample Mining

The test mining and bulk sample program at the Property was designed to refine the mining technique and confirm parameters for full-scale operations. The bulk sample mining allowed viewing of the Elk Gold Project gold-bearing quartz-pyrite vein system in three dimensions. Adding to what was interpreted previously from diamond drill core is the understanding that the mineralized system is a series of anastomosing veins and gouges centred around shear structures which can change on a scale of just a few metres. The main vein of interest (1300) was seen to persist to within a metre of the RBF dyke/fault. Drilling had previously not been able to confirm this continuity, affecting confidence in the block model in this area. Pit 2 mined out a portion of the dyke/fault, allowing for a more accurate measurement of strike and dip and width.

A post-mining reconciliation study examined drill results, block, and polygonal model estimates, plus mining results with the following conclusions:

- Drilled vein intercepts have significantly lower gold grades than that of the mined vein surrounding the drilling pierce point.
- This grade difference negatively affected both the block and polygonal resource model estimates.
- Plates and pile sample results were 11% greater than the settlement results for both gold grade and contained ounces; the settlement tonnage was adopted as actual.
- Block model estimates, which were based on a thicker-than-actual mining width at a much lower cut-off grade, suffered from changes to the mining method from that envisioned in the original geological model; in comparison to settlement results tonnage was overestimated by 270%, grade was underestimated by 72%, and contained ounces underestimated by 23%.
- The polygonal model more closely predicted actual mined tonnages, grades, and contained ounces of gold, although it overestimated actual tonnage by 8%, overestimated grade by 4%, and overestimated contained ounces by 12%.
- By applying a retroactive 20% dilution at zero grade and capping input grades at 100 g/t Au the polygonal resource estimate was recalculated to be within 1% of the tonnage and 2% of the actual mined grade. While the 20% dilution is thought to be a realistic number, the 100 g/t grade cap, at the 97th percentile, may change in future studies depending upon input grades.

25.3 Resource Estimation

The Mineral Resource is tabulated in Table 25-1. The Resource is divided into pit-constrained and underground portions. The pit-constrained resource is stated at a base case cut-off grade of 0.5 g/t Au and the underground resource is stated at a base case cut-off grade of 5.0 g/t Au. Ounces have been rounded to the nearest hundred and tonnes have been rounded to the nearest thousand.

Table 25-1: Elk Gold Mineral Resource Estimate Summary

Elk Gold Pit-Constrained Resources					
Cutoff Au Eq 0.50 g/t	**Tonnes**	**Au Equivalent (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	380,000	6.2	6.2	6.2	75,000
Indicated	2,111,000	4.8	4.7	8.5	319,000
Measured + Indicated	2,491,000	5.0	4.9	8.1	394,000
Inferred	259,000	4.6	4.5	9.6	37,000
Elk Gold Underground Resources					
Cutoff Au Eq 5.0 g/t	**Tonnes**	**Au Equivalent (g/t)**	**Au Capped (g/t)**	**Ag Capped (g/t)**	**AuEq Ounces**
Measured	5,000	7.48	7.42	5.70	1,000
Indicated	203,000	9.03	8.76	22.84	59,000
Measured + Indicated	208,000	8.99	8.73	22.43	60,000
Inferred	195,000	9.21	8.97	20.27	58,000

Notes: CIM definitions were followed for classification of Mineral Resources
Mineral Resources are not Mineral Reserves and have not demonstrated economic viability
Results are presented in situ and undiluted
Mineral resources are reported at a cut-off grade of 0.5 g/t Au for pit-constrained resources and 5.0 g/t for underground resources
The number of metric tons was rounded to the nearest thousand and the number of ounces of gold to the nearest hundred.
The Resource Estimate includes both gold and silver assays. The formula used to combine the metals is: AuEq = ((AU_Cap*41.8*0.96)+(Ag_Cap*0.55*0.86))/(41.8*0.96)
Effective date of the resource estimate is July 3, 2020.

25.4 Airborne Survey

An airborne magnetometer and radiometric survey flown over the central portion of the Property has provided valuable structural and geological information. A suite of filters and derivatives applied to the aeromagnetic gridded data, including multiscale edge detection, and automated grid 'fractal' analysis to has helped to identify regions prospective for gold mineralization based upon structural complexity as inferred from magnetics being indicative of significant faults or shears. Known zones of mineralization correspond well with CET analysis maps and of interest is that several untested zones also exist on the Property. Superimposing soil geochemical anomalies and known vein traces together with follow-up ground study, may well produce additional exploration targets.

The survey has shown the geology to be more complex than originally mapped, and multiple intrusive phases containing variable magnetic minerals the likely difference. Additional field mapping using the magnetic signatures as a guide will help to clarify the Property geology, though mineralized quartz vein emplacement does not seem to be affected by intrusive rock type.

Additional analysis and interpretation of the magnetic and radiometric data is available through 3-D inversions to identify discrete geometry of magnetic bodies. It is recommended that 3-D inversion modelling of the entire aeromagnetic dataset be conducted to identify contact zone dips.

25.5 Preliminary Economic Analysis

A PEA was carried out on the Elk Gold Project. This PEA included mineralized material classified as Inferred. Inferred material is considered too speculative geologically to have economic considerations applied to it and there is no certainty that the preliminary economic assessment will be realized. There is also no guarantee that Inferred resources will be converted to Measured or Indicated resources in the future.

This PEA envisioned a conventional open pit mine which begins as a toll milling operation, where mineralized material is hauled off-site for processing for Years 1 to 3. Beginning in Year 4 of the mine plan, it is assumed that a 900 t/d mill will be constructed on-site and that the mining rate will increase accordingly. The total resulting mine life is 10 years in which 2.4 Mt of mineralize material is processed at a grade of 5.3 g/t au and 8.8 g/t ag. The mine plan includes 83.6 Mt of non-mineralized rock resulting in a total strip ratio of 34.9 w:o.

The economic analysis from the PEA concluded that the proposed Elk Gold project appears to be a potentially economic open pit mine with an after-tax NPV (5%) of $190.8 million.

26 RECOMMENDATIONS

The following recommendations split the costs of additional work into two phases to mitigate the risk of raising financial resources.

26.1 Recommendations for Phase I Program

26.1.1 Siwash North Resource Drill Program

High-grade gold mineralization is known to occur in patches within the main Siwash Vein (1300) down dip of Pit 1 and the vein is known to exist and carry gold to at least 350 m down dip. Several drill holes would be required to determine if high-grade gold exists at depth to support further development. The drill holes would need to be at least 300 m each to test the vein down dip of intersections from previous programs.

The estimated budget for Siwash North underground resource drilling program is $350,000.

26.1.2 Lake Zone Drill Program

The Lake zone is an exploration target which has 30 historical drill holes. It is recommended to drill additional drill holes filling gaps in the existing drill data and stepping out to test the extension of the known mineralization. The objective of the drill program is to improve the geological understanding of the zone to a sufficient level to estimate a maiden resource.

The Lake Zone in-fill and step out drilling program is estimated to cost $150,000.

26.2 Recommendations for Phase II Program

26.2.1 Review Wireframe Model

It is recommended to carry out a detailed review of the resource wireframe model to optimize the drill-hole intercepts and improve the consistency of the geological interpretation of the drill-hole data. Although this review is not anticipated to have a material impact on the modelled resources, it may improve the resource estimate by better reflecting the available geological information.

The cost of reviewing the wireframe model is $20,000.

26.2.2 Complete a Pre-Feasibility Study

A PFS is recommended to provide information to shareholders concerning the economic viability of the Elk Gold Project. The PFS would include Mineral Reserves, process plant design for on-site processing, and an underground assessment of potential mining methods.

A PFS is estimated to cost $250,000.

26.2.3 Metallurgy

Recommendations for additional metallurgical testwork are listed below:

- Phase I—Three global composites be prepared and used for flowsheet optimization, representing the proposed mine plan for Year 0 to 1, 1 to 3, and 3 to end of mine. The composites will be subjected to mineralogical analysis, comminution testwork, gravity concentration, flotation tests including locked cycle tests, and settling and filtering assessments.
- Phase II—Twenty composites representing discrete continuous intervals of mineralization be used to assess variability in the deposit. The samples will undergo mineralogical analysis as well as comminution testwork. The optimized flowsheet and parameters established in the Phase I program will be used as the basis for the flotation testwork in Phase II, to establish metallurgical performance.

The samples will be evaluated for gold and silver with an estimated test program cost of $350,000.

26.2.4 Geophysical Analysis

An airborne magnetic and radiometric survey was carried out over the Elk Gold Project in 2019 and the initial assessment of the results included encouraging signatures which indicated that areas outside of the known mineral occurrences may be prospective for additional discoveries. It is recommended that additional analysis be carried out the geophysical data collected in 2019 to better refine the results and propose area's for field work follow-up.

The cost of the additional geophysical analysis is estimated to be $35,000.

26.2.5 Elusive Creek Zone Drill Program

The Elusive Creek zone consists of a strong gold-in-soils anomaly which was followed up by surface trenching. The surface exploration completed appears to indicate that the Elusive Zone is possibly a different style of deposit compared to the Siwash North resource and that it could be more of a massive mineralized zone rather than discreet mineralized veins. It is recommended to follow-up the surface exploration work in the Elusive Creek zone with a series of exploratory diamond drill holes to better develop the geological understanding of the zone.

The estimated cost of the Elusive Creek zone exploration program is $150,000.

26.2.6 Gold Creek Zone Drill Program

The Gold Creek zone is an exploration target with 24 existing diamond drill holes with encouraging assay results. The zone is located immediately south of the Siwash North resource, within the mining lease. It is recommended to carry out an in-fill and step-out drill program in the Gold Creek zone to fill gaps in the existing geological data and test the extension of the mineralization. The objective of the program is to improve the geological confidence in the zone to be able to estimate an Inferred resource.

The estimated cost of the Gold Creek zone exploration program is $650,000.

26.2.7 South Showing Drill Program

The South Showing is an exploration target with 40 existing diamond drill holes with encouraging assay results. The zone is located approximately 5km south of the Siwash North resource, within the mining lease. It is recommended to carry out an in-fill and step-out drill program in the South Showing zone to fill gaps in the existing geological data and test the extension of the mineralization. The objective of the program is to improve the geological confidence in the zone to be able to estimate an Inferred resource.

The estimated cost of the South Showing exploration program is $495,000.

26.2.8 Geotechnical Drilling

Prior to future mining programs, geotechnical drilling would be required to determine the wall slopes for a mining open pit and underground development. While the selection of the number and locations of holes has not been determined, it is recommended to allow $300,000 for this study. Note that costs would be less if geotechnical drilling were incorporated into the main drilling program.

26.2.9 Summary of Costs

The work programs recommended in Sections 26 have individual costs associated. These costs are summarized for Phases I and II (Table 26-1).

Table 26-1: Estimated Cost of Recommended Work Programs

Item	Cost ($ million)
Phase I Program	
Siwash North Resource Drill Program	0.350
Lake Zone Drill Program	0.150
Subtotal of Phase I	*0.500*
Phase II Program	
Review Wireframe Model	0.020
Complete an PFS Study	0.250
Metallurgical Testwork	0.350
Geophysical Analysis	0.035
Elusive Creek Drill Program	0.150
Gold Creek Drill Program	0.650
South Showing Drill Program	0.495
Geotechnical Drilling	0.150
Subtotal of Phase II	*2.100*
Total	**2.600**

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28 CERTIFICATE OF AUTHORS

28.1 Antonio Loschiavo, P.Eng.

I, Antonio Loschiavo, P.Eng., of Vancouver, BC, as an author of this Technical Report titled *NI 43-101 Technical Report Preliminary Economic Assessment on the Elk Gold Project, Merritt, British Columbia, Canada,* with an Effective Date of July 3, 2020 (the "Technical Report"), prepared for Gold Mountain Mining Corp., a Freeform Capital Partners Inc. (the "Issuer") do hereby certify that:

- I am the President of AKF Mining Services Inc. in Vancouver, BC, Canada; c/o Unit 520, 6362 Fraser Street, Vancouver, BC, V5W 0A1.
- I am a graduate of University of British Columbia with a B.A.Sc. in Mining & Mineral Process Engineering, 1998.
- I am a member in good standing with the Association of Engineers and Geoscientists of British Columbia, License #39436.
- I have practiced my profession continuously since 1998. My relevant experience includes more than 20 years in mining operations, project studies, and engineering. I have held senior mine planning and technical mine positions in Canada. I have worked as a consultant for over 10 years and have performed, resource, reserves, optimizations, mine planning, scheduling, cost estimation, and economic analysis work for a significant number of projects, throughout Canada, USA, Europe, Russia, China, and South America.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience I fulfill the requirements to be a "qualified person: for the purposes of NI 43-101.
- I have visited the Elk Gold Project site on multiple occasions and my last visit was June 21, 2019.
- I am responsible for Sections 15, 16, and 18 to 22, and Sections 1.1, 1.9, 1.10, 1.12, 1.13, 1.15, 1.16, 1.17, 25.5, 26.1, 26.2.2, 26.2.8, 26.2.9, and 27 of this Technical Report.
- I am independent of the Issuer and related companies applying all the tests in Section 1.5 of the NI 43-101.
- I have had prior involvement with the Property providing engineering support for the bulk-sampling program in 2012 and 2014.
- I have read NI 43-101 and confirm this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the Effective Date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated in Vancouver, British Columbia, this 21st day of September 2020.

Original Signed and Sealed

Antonio Loschiavo, P.Eng.

28.2 Robert G. Wilson, P.Geo.

I, Robert G. Wilson, P.Geo., of Langley, British Columbia, as an author of this Technical Report titled *NI 43-101 Technical Report Preliminary Economic Assessment on the Elk Gold Project, Merritt, British Columbia, Canada,* with an Effective Date of July 3, 2020 (the "Technical Report"), prepared for Gold Mountain Mining Corp., a Freeform Capital Partners Inc. (the "Issuer") do hereby certify that:

- I am an independent geologist residing in the Township of Langley, BC.
- I obtained a Bachelor of Science degree in Geology in 1976 from The University of British Columbia, Vancouver, BC.
- I am Registered Professional Geoscientist (License #19688) in good standing with the Association of Engineers and Geoscientists of British Columbia.
- I have been continuously practicing my profession as a geologist since 1977 for a variety of major and junior companies.
- I have read the NI 43-101 *Standards of Disclosure for Mineral Projects* and Form 43-101F1, and that this Technical Report has in part been prepared by me, in compliance with the foregoing Instrument and Form.
- I am independent of the issuer applying as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.2 to 1.4, 2 to 12 (except 10.4), 23, 24, 25.1, 25.2, and 26.2.4 to 26.2.7 of this Technical Report.
- This report is based on my knowledge of the Elk Gold Project gained from my capacity as an employee with the titles Senior Geologist and Project Manager for Gold Mountain Mining Corp. from June 2011 to September 2015, and a review of proprietary and published reports and maps on the subject property and surrounding area.
- I am not aware of any material fact or material change with respect to the subject matter of the report, which is not reflected in the report and by which, the omission to disclose would make the Technical Report misleading.

Dated in Vancouver, British Columbia, this 21st day of September 2020.

Original Signed and Sealed

Robert G. Wilson, P.Geo.

28.3 Gregory Z. Mosher, P.Geo.

I, Gregory Z. Mosher, P.Geo., of North Vancouver, British Columbia, as an author of this Technical Report titled *NI 43-101 Technical Report Preliminary Economic Assessment on the Elk Gold Project, Merritt, British Columbia, Canada,* with an Effective Date of July 3, 2020 (the "Technical Report"), prepared for Gold Mountain Mining Corp., a Freeform Capital Partners Inc. (the "Issuer") do hereby certify that:

- I am a Principal Geologist with Global Mineral Resource Services with a business address at #603 – 131 East Third Street, North Vancouver, BC, Canada, V7L 0E3.
- I am a graduate of Dalhousie University (B.Sc. Hons., 1970) and McGill University (M.Sc. Applied, 1973).
- I am a member in good standing with the Association of Engineers and Geoscientists of British Columbia, License #19267. My relevant experience with respect to gold deposits includes over 30 years of exploration and evaluation of such deposits. In addition, I have been performing resource estimates of gold deposits since 2005.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I have been continuously practicing my profession as a geologist since 1973 for a variety of major and junior companies.
- I have read the NI 43-101 *Standards of Disclosure for Mineral Projects* and Form 43-101F1, and that this Technical Report has in part been prepared by me, in compliance with the foregoing Instrument and Form.
- I am independent of the issuer applying as defined in Section 1.5 of NI 43-101.
- I have conducted a site visit of the Property on June 21, 2019 for a period of one day.
- I am responsible for Sections 1.5, 1.8, 10.4, 14, 25.3, and 26.2.1 of this Technical Report.
- I am not aware of any material fact or material change with respect to the subject matter of the report, which is not reflected in the report and by which, the omission to disclose would make the Technical Report misleading.

Dated in Vancouver, British Columbia, this 21st day of September 2020.

Original Signed and Sealed

Gregory Z. Mosher, P.Geo.

28.4 Kelly McLeod, P.Eng.

I, Kelly McLeod, P.Eng., of Vancouver, British Columbia, as an author of this Technical Report titled *NI 43-101 Technical Report Preliminary Economic Assessment on the Elk Gold Project, Merritt, British Columbia, Canada,* with an Effective Date of July 3, 2020 (the "Technical Report"), prepared for Gold Mountain Mining Corp., a Freeform Capital Partners Inc. (the "Issuer") do hereby certify that:

- I am a Principal Metallurgist with K-Met Consultants Inc with a business address at 1181 Cordova Street, Vancouver, BC, Canada, V6C 0A1.
- I am a Professional Metallurgical Engineer registered with the EGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelor of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984 and have worked for the last 14 years consulting in the mining industry in metallurgy and process design engineering.
- I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101.
- I have not visited the Elk Gold Project site.
- I am responsible for Sections 1.7, 1.11, 13, 17 and 26.2.3 of this Technical Report.
- I am not aware of any material fact or material change with respect to the subject matter of the report, which is not reflected in the report and by which, the omission to disclose would make the Technical Report misleading.

Dated in Vancouver, British Columbia, this 21st day of September 2020.

Original Signed and Sealed

Kelly McLeod, P.Eng.

29 DATE AND SIGNATURE PAGE

This Technical Report titled *NI 43-101 Technical Report Preliminary Economic Assessment on the Elk Gold Project, Merritt, British Columbia, Canada*, with an Effective Date of July 3, 2020, was written by the following Qualified Persons and contributing authors.

Qualified Person	Signature	Date
Antonio Loschiavo, P.Eng.	*Original Signed and Sealed*	September 21, 2020
Robert G. Wilson, P.Geo.	*Original Signed and Sealed*	September 21, 2020
Gregory Z. Mosher, P.Geo.	*Original Signed and Sealed*	September 21, 2020
Kelly McLeod, P.Eng.	*Original Signed and Sealed*	September 21, 2020