Preliminary Economic Assessment, NI 43-101 Technical Report, for the Scottie Gold Mine Project in Northwestern British Columbia, Canada

PREPARED FOR

Scottie Resources Corp.

EFFECTIVE DATE: OCTOBER 28, 2025

719-25188-RPT-001

PREPARED BY:

SUE BIRD, P.ENG.

DAMIAN GREGORY, P.ENG.

HASSAN GHAFFARI, M.A.SC., P.ENG.

JIANHUI (JOHN) HUANG, PH.D., P.ENG.

Table of Contents

scottie RESOURCES CORP

scottie
RESOURCES CORP

scottie RESOURCES CORP

scottie
RESOURCES CORP

LIST OF TABLES

LIST OF FIGURES

UNITS OF MEASURE

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
bank cubic metres	bm^3
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
centimetre	cm
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic yard	yd^3
Coefficients of Variation	CVs
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	dBA
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (United States)	USD$
dollar (Canadian)	CAD$
dry metric ton	dmt
foot	ft or ′
gallon	gal
gallons per minute (US)	gpm
gauge	ga
gigajoule	GJ
gigapascal	GPa
gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp

hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	in. or "
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
less than	<
litre	L
litres per minute	L/m
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
metric ton (tonne)	t
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
Neutron	N

ounce	oz
pascal	Pa
centipoise	mPa·s
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
tonne (1,000 kg)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
troy ounce	t oz
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt
year (annum)	a

1.0 SUMMARY

Scottie Resources Corp. (Scottie) retained Tetra Tech Canada Inc. (Tetra Tech) to prepare a National Instrument 43-101 (NI 43-101) Preliminary Economic Assessment (PEA) Technical Report (the Report) for the Scottie Gold Mine Project (the Project) located in northwestern BC, Canada. The pertinent PEA News Release was disseminated on October 28, 2025, by Scottie, which is available on Scottie's corporate website. (https://scottieresources.com/news/2025/)

The effective date of this Technical Report is October 28, 2025, and the effective date of the Mineral Resource Estimate (MRE) is February 2, 2025.

1.1 Property Description

The property is found within the Skeena Mining Division and the claim boundaries were obtained from government claim maps. The crown granted claims, and mineral tenures are entirely owned by Scottie Resources. The claims/crown grants are subject to a 2% Gross Production Royalty held by Franco-Nevada. No other royalty or encumbrance exists on the claims.

Figure 1-1: Location of the Scottie Gold Mine Project (Scottie, 2025)

1.2 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.2.1 Accessibility and Climate

The well-maintained Granduc Mine Road passes through the claim boundaries, which provides access to the northeastern areas of the Property. The climate is classified as humid continental, with an average annual temperature of 6.1˚C, and an average yearly precipitation 1866 mm.

1.2.2 Local Resources and Infrastructure

The town of Stewart, BC, has a population of approximately 500 and is host to an ice-free deep-water port, a paved airstrip, as well as several stores that supply basic amenities. The city of Terrace and town of Smithers are both approximately four hours away by vehicle from Stewart. Both provide access to commercial airports along with most other services required to support mineral exploration and mining projects.

1.2.3 Physiography

The property is located within the Boundary Ranges of the Coast Mountains. The elevation on the property ranges from a low of 700 m above sea level (asl) to a high of 2,126 m (asl), found at the peak of Summit Mountain.

1.3 History

1.3.1 Scottie Mine Area

The first exploration undertaken in the Stewart area was in 1898 by prospectors exploring the area on their way to the Klondike gold rush. The initial discovery of gold-bearing veins in the Scottie Gold Mine area was in 1928. The property was optioned to Premier Gold Mining Company (“Premier”) in 1931, with surface sampling and trenching operations revealing ore-grade mineralization in two zones along-strike lengths of 85 to 350 feet.

Between 1946-1955, Morris Summit Gold Mines Ltd. (“Morris Summit”) completed diamond drilling, lateral development work and raised development from a portal developed at the 3,000’ level. These efforts helped identify four mineralized shoots including the McLeod East Zone (now part of the M Zone).

In 1955, they re-sampled the historical workings and diamond drillholes to substantiate the historical findings, then followed this up in 1956 with surface prospecting and geophysics, which outlined several more gold-bearing veins, after which the property remained idle until 1978 when the controlling interest of Morris Summit was acquired by D. A. McLeod and Associates of Vancouver (“McLeod Group”). The McLeod Group developed an access road through to the 3,000’ level adit in 1978 and completed 3,058’ of diamond drilling at the M Zone between 1978 and 1979.

A feasibility study, completed in 1980 by McLeod and based on a gold price of $660/ounce, recommended placing the property into production at 200 tons/day. The Scottie Gold Mine was put into production on October 1, 1981, and continued for about 3.5 years until February 18, 1985.

The mine was placed into receivership by the Royal Bank of Canada in February 1985, with a re-organization of Scottie Gold Mines resulting in the formation of a new company – Royal Scot who resumed exploration on the property in 1987.

Since the shutdown of the mine the Scottie Gold Mine Project area has been explored various exploration companies including Royal Scott, Tenajon Resources, Arkaroola, Seeker Resources, Jayden Resources, Red Eye, and Rotation Minerals, which changed names to Scottie Resources in 2019.

1.3.2 Blueberry and Bend Area

In 1983, Esso Resources Canada Ltd. (“Esso”) carried out prospecting within the Tide Lake mineral reserve, locating a massive sulphide occurrence along the mine road that was named the Bend Vein showing.

In January of 1984, the Summit Group claims, comprising the Bow 1, Wow 1, Wow 2, and Wow 4 claims, was acquired through staking by a joint venture between Esso (50%) and Scottie Gold Mines Ltd. (50%). Holes drilled at Bend intercepted the structure and the Bend Vein was intercepted over 60 m of strike length with an average vein width of 1.5 m. The Blueberry Vein was also exposed over 90 m.

In 1989, Homestake purchased the assets of Esso Resources and in 1990 Homestake Mining (Canada) Ltd. entered into an agreement with Tenajon for exploration of the Bow 1 claim outside of the Bend Vein area. Tenajon conducted work programs, including trenching, sampling, and drilling until 2008. The property was acquired by Rotation in 2017.

1.4 Geologic Setting and Mineralization

The Scottie Gold Mine Property lies above the volcano-sedimentary rocks of the upper Stuhini and lower Hazelton Groups. The rocks in this area have undergone several generations of separate intrusive events, with dykes and stocks of variable ages and compositions seen throughout. During the Cretaceous period, east-northeast compression caused the development of north-northwest trending upright folds, resulting in the formation of the Summit Mountain anticline with its Upper Triassic core exposed on the western side of the property.

The Blueberry Contact Zone target comprises the north-south oriented andesite-siltstone contact and numerous moderately northwest dipping veins. The north end of the Blueberry Contact Zone is offset by an east-west, dextral fault and can be characterized by north and south portions.

Gold mineralization on the property is thought to be of intrusion-related gold deposit style. Anomalous gold values occur in shoots that are hosted in veins and replacement zones, with the highest grades typically correlated with increased sulphide content. Base metal and silver values are variable with a few areas on the property with polymetallic veining producing strongly anomalous silver, copper, lead, and zinc values. However, in gold rich areas such as Blueberry Contact Zone and Scottie Gold Mine area, base metal and silver values are only slightly to moderately elevated but form a much broader footprint than gold mineralization.

1.5 Exploration

Property-wide geological mapping, 2D induced polarization surveys, airborne mag, and EM surveys have been done since 2019. In addition, 3D DC-resistivity and induced polarization (DCIP) surveys have been completed at the Blueberry and Domino zones, along with 1,560 m of borehole TEM surveying carried out at the Scottie Gold Mine.

Several prospecting programs have also been completed from 2021 through 2024 at the Blueberry, Gulley, Serac, C, D and E Zones, Scottie Mine, High-Grade Float Zone, Golden Buckle, and Scottie East Zones.

1.6 Drilling

Scottie Resources operated several small diamond drilling campaigns on the Scottie Gold Mine Property. From 2016 - 2019 the company was called Rotation and in 2019 had a name change to Scottie Resources Corp. The bulk of drilling has been done with skid- and helicopter-portable diamond drilling rigs, with methods and results discussed in Section 10.

1.7 Sample Preparation Analysis and Security and Data Verification

Sample preparation, security, and QAQC has been documented and analyzed as presented in Section 11 for all years with information available. Historical data with no available information has been validated using the recent data.

1.8 Mineral Resource Estimate

The Scottie Project total MRE includes the Scottie and Blueberry deposits, with the Blueberry containing satellite deposits called the Bend and Gulley zones. The MRE is summarized in Table 1-1 for the base case cut-off grades. Mineral Resources were estimated using the 2019 CIM Best Practice Guidelines and are reported using the 2014 CIM Definition Standards.

The resource uses pit shells to constrain resources at the Blueberry deposits and potentially minable underground shapes at varying cutoff grades to define the underground resource below the Blueberry pit and for the Scottie Mine underground resources. The current estimate uses a metal price of US $2,000/oz gold price, recoveries, smelter terms, and costs, as summarized in the notes to Table 1-1. Metal prices have been chosen based partially on three-year trailing averages and industry standard pricing currently used for resource estimates.

The base case cut-off grade for open pit mining is 0.70 g/t Au and 2.5 g/t Au for underground resources, which more than covers the Processing + G&A for the open pit mining and covers costs of Processing + G&A + underground development costs for the underground resource.

These mineral resource estimates are Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Sue Bird, P.Eng., QP, is of the opinion that issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work. These factors may include environmental permitting, infrastructure, sociopolitical, marketing, or other relevant factors.

As a point of reference, the in-situ gold is inventoried and reported by intended processing method.

The resource estimate for the PEA is based on inferred resources as stated in the February 2025 Resource Estimate for the Scottie Gold Mine project. Certain mining factors have been applied to this resource estimate to generate diluted resources using a conceptual mine plan for the PEA. The February 2025 Resource Estimate is summarized below:

Table 1-1: Mineral Resource Estimate for the Scottie Gold Mine Project

Blueberry Pit Resource					
Source	**Cutoff Au (g/t)**	**Tonnage (ktonnes)**	**Au (g/t)**	**NSR ($CDN)**	**Au Metal (kOz)**
Blueberry Pit (Inferred)	0.25	2,887	2.06	156.04	191
	0.3	2,712	2.17	164.69	190
	0.5	2,114	2.68	202.51	182
	0.7	**1,707**	**3.17**	**239.73**	**174**
	1	1,323	3.85	290.19	164
	2.5	600	6.61	492.83	128
	5	273	10.35	755	91
Total Underground Resource					
Source	**Cutoff Au (g/t)**	**Tonnage (ktonnes)**	**Au (g/t)**	**NSR ($CDN)**	**Au Metal (kOz)**
Blueberry and Scottie Mine Underground (Inferred)	**2.5**	**1,897**	**8.66**	**678.51**	**528**
	3	1,704	9.33	731	511
	3.5	1,549	9.94	778.78	495
	4	1,404	10.59	829.04	478
	4.5	1,269	11.26	881.69	459
	5	1,143	11.98	937.99	440
	10	520	18.05	1,413.75	302
Total	**varies**	**3,604**	**6.06**	**470.69**	**703**

Notes to the 2025 Resource Table:

1. Resources are reported using the 2014 CIM Definition Standards and were estimated using the 2019 CIM Best Practices Guidelines, as required National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”).
2. The base case MRE has been confined by “reasonable prospects of eventual economic extraction” shape using the following assumptions:
   - Metal price of US $2000/oz gold
   - Metallurgical recovery of 90% gold
   - Payable metal of 99% gold in doré
   - Forex of 0.74 $ US:$ CDN
   - Processing costs of CDN $24/tonne milled, which includes milling, transport, smelter treatment, refining, and General & Administrative (G&A) costs
   - Underground production cost of CDN $78/tonne, and underground development costs to be CDN $90/tonne, for a total underground mining cost of CDN $168/tonne
   - Open pit mining costs of CDN $3.00/tonne for mineralized and waste material
   - 45-degree pit slopes
   - The 130% price case pit shell is used for the confining shape with elevation adjustment of the main Blueberry pit for the underground resource.
3. The resulting net smelter return is NSR = Au g/t* CDN $98.60 / g * 90% recovery rate
4. Numbers may not add due to rounding
5. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the estimated mineral resources will be converted into mineral reserves.

It is noted that the prices and costs used for the resource estimate are not identical to the updated values used for the mining and cash flow calculations. The costs and Au price used for the resource reflect those considered reasonable at the effective date of the resource estimate. A check has been done on these values compared to the current values and it is found that the resource is somewhat conservative on price due to the 3-year average gold price that was used in the 2025 MRE report, whose effective date was in Q1 2025, compared to the current 3-year average gold price used in this PEA report and the gold price has been rising in the nine months since. This has resulted in a similar cutoff for open pit mining and somewhat higher cutoff for underground than that used for the mining study and cashflow.

The Qualified Person is of the opinion that issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work. These factors may include environmental permitting, infrastructure, sociopolitical, marketing, or other relevant factors.

1.9 Mining Methods

The project is planned as a combined open pit and underground mining operation. Open pit development is expected to use a conventional truck-and-shovel method, whereas underground development will be based on longitudinal longhole stoping. The processing plant is designed to process the plant feed at a nominal throughput of 1,000 t/d, with a LOM average of 900 t/d. This has been assumed for the combined operation.

Mine planning was completed using the mineral resource model provided by MMTS, including the Mineral Resource Inferred classification. There is no Mineral Reserve estimate for the project as it is currently at a PEA stage.

For the base case scenario, run-of-mine (ROM) plant feed will be stockpiled and subsequently processed using an ore sorter. The ore-sorter concentrate will be shipped directly to overseas markets. This mining and processing strategy has been applied to both the open pit and underground components and forms the basis of the PEA.

The Blueberry deposit was assessed as a combination of open pit and underground mining. The Scottie deposit, which was previously mined using shrinkage stoping ending 40 years ago, was assessed as underground only.

The mine schedule for both open pit (OP) and underground (UG) is shown in Figure 1-2 and Figure 1-3.

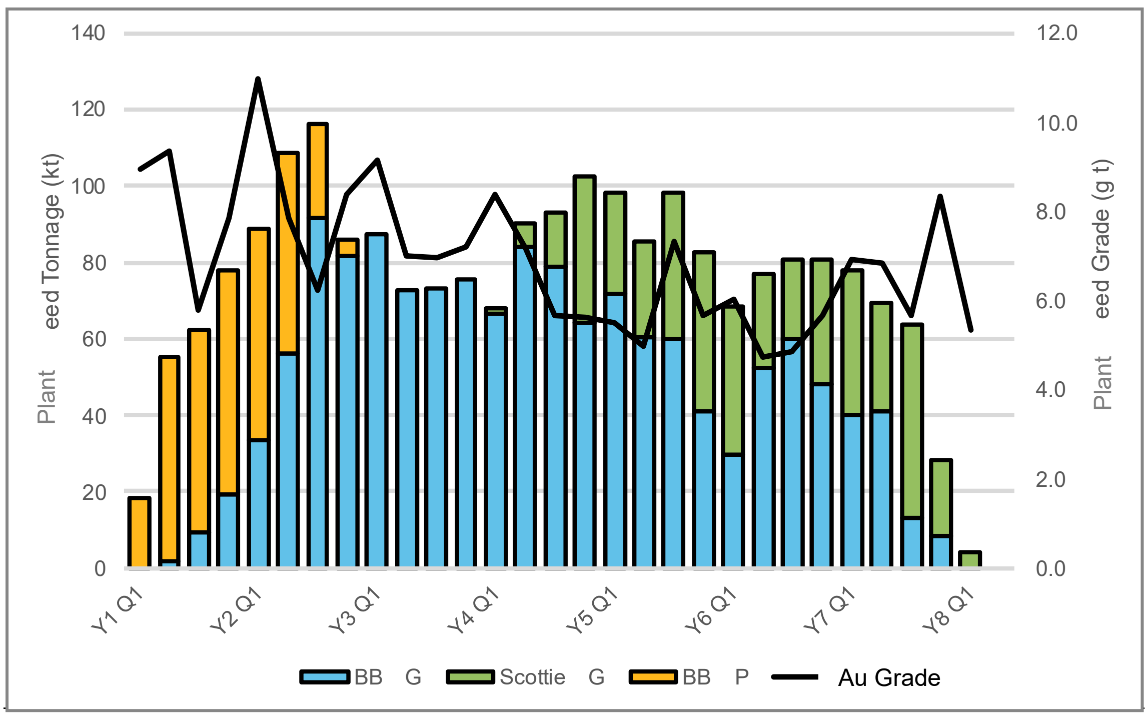


Figure 1-2: Quarterly Plant Feed

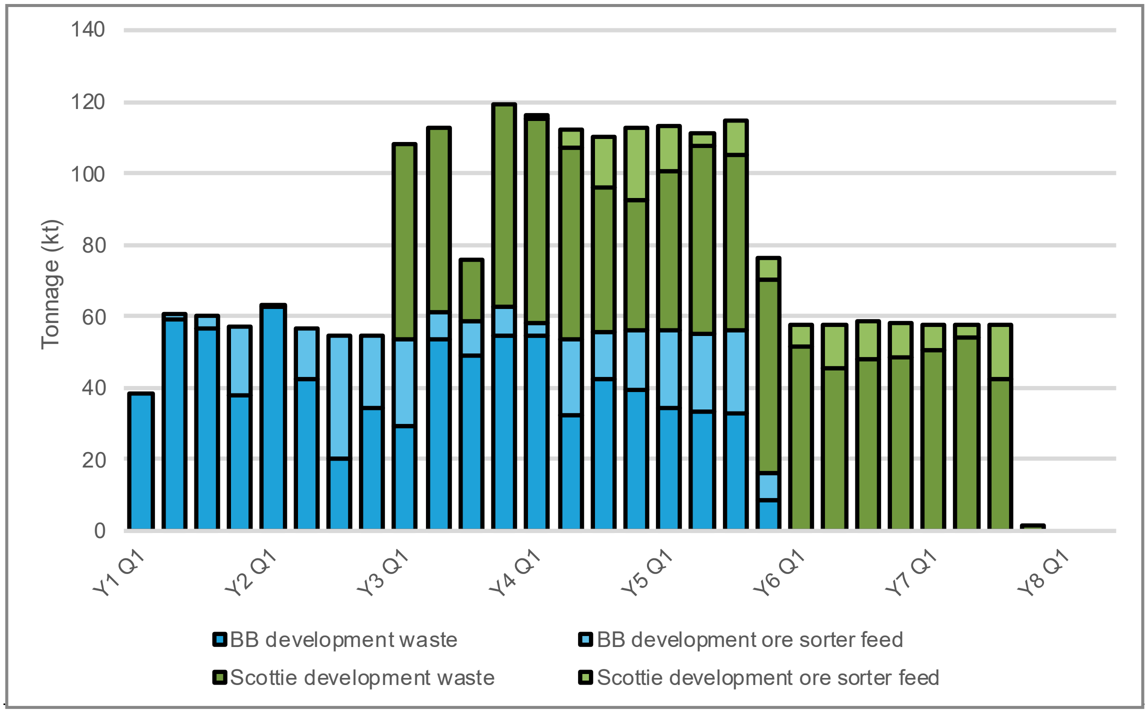


Figure 1-3: Quarterly Underground Development

The total combined mining inventory, inclusive of Mineral Resource classified as Inferred, is shown in Table 1.2.

Table 1-2 Combined Potential Ore-Sorter Feed

Phase	Rock (Mt)	Ore-Sorter Feed (Mt)	Waste (Mt)	Au (g/t)	Au Met to Mill (kOz)	Au Met Recovered (kOz)
Total OP	5.86	0.32	5.54	7.71	79	76
Total UG	3.60	1.87	1.73	6.72	404	385
Total	9.46	2.19	7.27	6.87	483	460

1.10 Metallurgical Testing

Since 1980, numerous phases of metallurgical testing have been conducted on Scottie samples to investigate the gravity-recoverable gold content, the cyanide leaching performance of head composite samples, and the response of gravity tailings to flotation and cyanidation processes. These early studies laid the groundwork for understanding the mineralization's behavior under conventional recovery methods. Test work conducted in 2023 was also focused on conventional recovery methods.

Beginning in 2024, the focus of metallurgical work shifted toward plant feed pre-concentration using ore sorting conducted during 2024/2025 and dense media separation (DMS) conducted in 2025. Some additional tests to characterize mineralogical characterization, such as gold distribution in various particle size fractions, and determine process related parameters, such as Bond crushing and grinding work indexes.

Further test work is recommended to determine most effective process for gold recovery from the mineralization.

1.11 Recovery Methods

The plant will be fed from three distinct mining areas: Blueberry Open-Pit (BBOP), Blueberry Underground (BBUG), and Scottie Gold Mine Underground (SGMUG). Mining will be conducted using conventional truck-shovel open-pit methods for BBOP, and longitudinal long-hole stoping for both BBUG and SGMUG. Run-of-mine (ROM) plant feed will be processed through single-stage crushing, operating in a closed circuit with a screen. The screen oversize products—coarse and fine fractions—will be treated using respective coarse and fine sorters to produce concentrates. The sorter concentrate products will be blended with the fine rejects produced from the screen in the crushing circuit and shipped as the final product.

The processing plant has been designed to process plant feed at a nominal throughput of 1,000 t/d, producing gold concentrate. The LOM average plant feed grade is estimated to be 6.86 g/t Au, and the anticipated average gold recovery will be 94.7%, including the fines blending with sorter concentrates. The LOM average annual concentrate production will be approximately 177,000 t/y at 11.5 g/t Au. The simplified overall process flowsheet is presented in Figure 1-4.

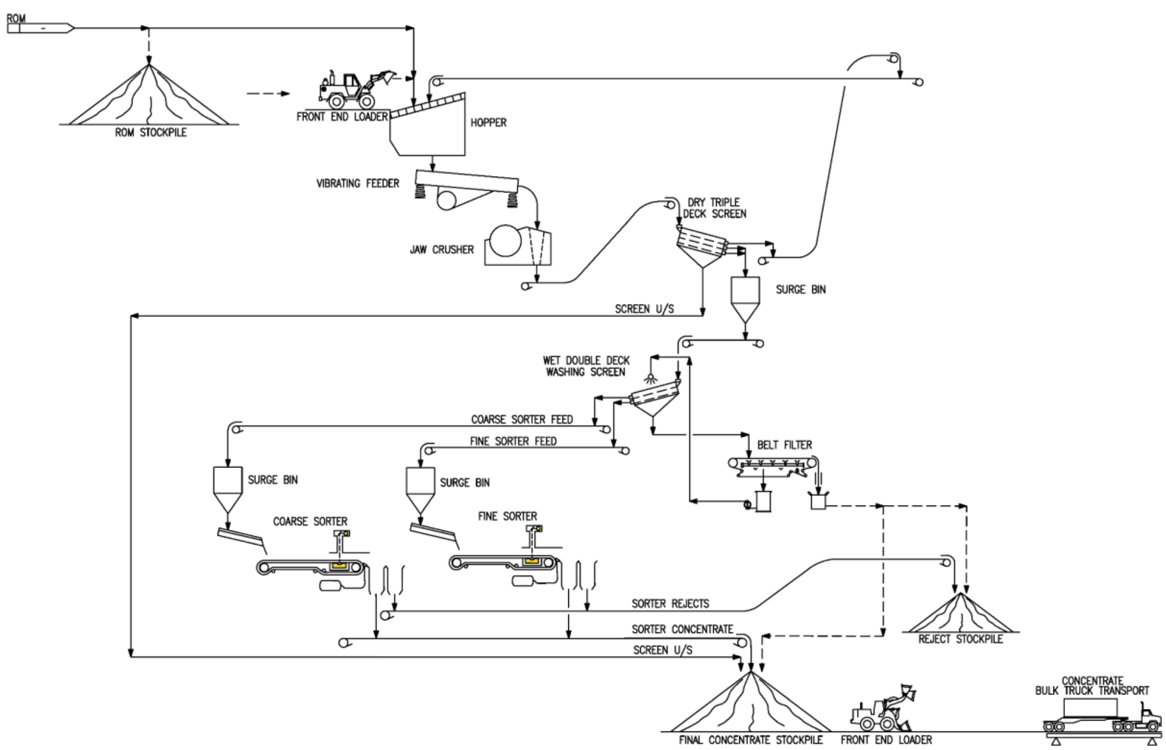


Figure 1-4: Simplified Overall Process Flowsheet (Tetra Tech, 2025)

1.12 Project Infrastructure

The Project will require the development of a number of infrastructure items. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and for efficient and convenient operation of the mine equipment fleet. Buildings will be equipped with high pitched roofs for efficient snow clearing.

The Project surface infrastructure will include the following:

- The existing internal access road network with future upgrades, realignment and new segments to facilitate efficient logistics on the Project site
- A sorting plant with sorter concentrate and reject stockpiles and product loadout
- A modular 160-person accommodation camp with arctic corridors
- A surface mobile equipment shop and warehouse with mine dry and offices
- A cold storage warehouse
- Detonator and Explosive Storage Magazines
- A surface maintenance laydown and storage area
- A power plant with a 13.8kV power distribution system
- Water supply and distribution
- Fuel storage and distribution
- A site communication system
- Waste management facilities
- A sewage treatment facility
- Surface water management structures
- Mine and sorted waste rock storage facility
- ROM storage facilities
- Avalanche mitigation structures

The overall site General Arrangement is present in Figure 1-5.

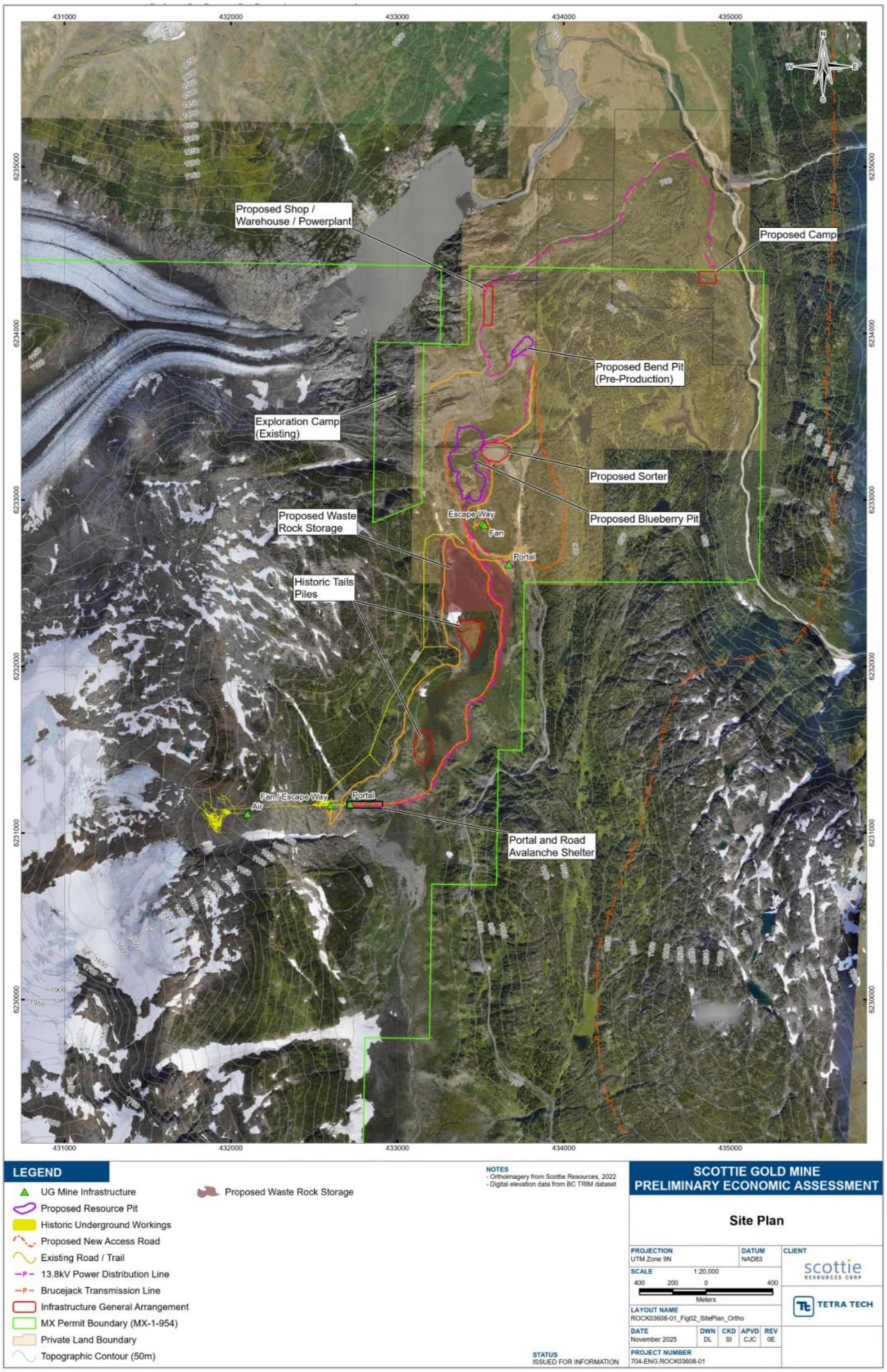


Figure 1-5: Site General Arrangement (Tetra Tech, 2025)

1.13 Capital Cost Estimate

The capital and sustaining capital costs for the Project have been estimated and are summarized in Table 1-2.

Table 1-2: Summary of Capital Costs

Description	Total Capital Cost (C$ million)
Initial Capital Costs	128.6
Sustaining Capital for LOM	76.7

All costs are reflected in 2025 Q3 Canadian Dollars unless otherwise specified. The expected accuracy range of the cost estimates is within ±35%. Where applicable, costs in this report have been converted from US Dollars to Canadian Dollars using a currency exchange rate of CAD1.00:USD0.72.

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $128.6 million. A summary breakdown of the initial capital cost is provided in Table 1-3.

Table 1-3: Initial Capital Cost Summary

Description	Initial Capital Cost (C$ million)
Direct Costs	
Mining Infrastructure	6.8
Site Preparation	5.0
Sorting Plant	26.2
On-Site Facilities: Camp, Power Plant Laboratory, and Other Facilities	28.5
Surface Mobile Equipment	7.8
Utilities, such as Fresh/Potable Water, Power Distribution and Waste Management	7.3
Water Management and Avalanche Control	3.0
Subtotal – Direct Costs	**84.6**
Indirect Costs	
Project Indirect Costs	23.7
wner’ s Costs	3.4
Contingencies	16.9
Total	**128.6**

1.14 Operating Cost Estimate

On average, the LOM on-site operating costs for the Project were estimated to be $185.40/t of plant feed. The operating costs are defined as the direct operating costs including mining, processing, site servicing, and G&A costs, including worker transport, freight costs, and catering costs. Table 1-4 and Figure 1-6 show the cost breakdown for various areas.

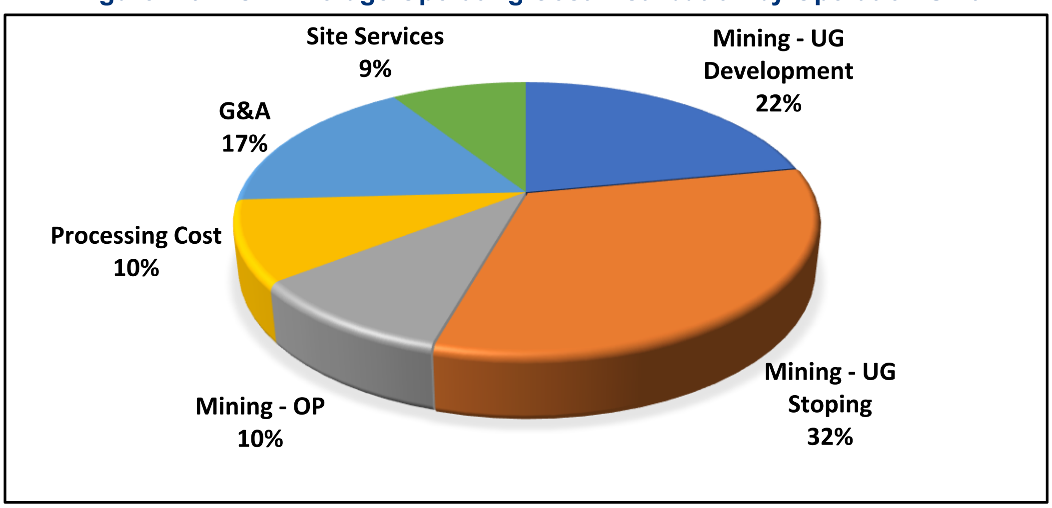
The cost estimates in this section are based upon the consumable prices and labour salaries/wages in Q3 2025 or based on the information from the Tetra Tech's database. Where applicable, costs in this estimate have been converted from US Dollars to Canadian Dollars using the currency exchange rate of CAD1.00:USD0.72. The expected accuracy range of the operating cost estimate is within ±35%.

Table 1-4: LOM Average Operating Cost Summary

Function	Operating Cost (C$/t Plant Feed)
Mining Cost - UG-Development	40.71
Mining Cost - UG-Stoping	60.13
Mining Cost - OP	18.59
Processing Cost	17.96
G&A	31.23
Site Services	16.76
Total Operating Cost	**185.38**

Note: 1. LOM average operating at 900 t/d, which is slightly different from the unit cost at the nominal process rate of 1,000 t/d. 2. G&A includes worker transport freight and catering costs.

Figure 1-6: LOM Average Operating Cost Distribution by Operation Unit



1.15 Economic Analysis

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The economic analysis of the Project has been derived from the inputs described in this report and performed on a 100% basis using Q3 2025 Canadian dollars, and its unlevered post-tax FCF has been discounted using mid year discounting at a rate of 5% per annum. Capital, operating, sustaining and closure costs, net profits interest (NPI) payments, BC Mineral tax, and Federal and Provincial income taxes are included in the financial analysis.

For the seven-year mine life and 2.19 Mt processing plant feed tonnage, 1.24 Mt sorted concentrate tonnage, and the foreign exchange rate of CAD1.00:USD0.72, the following investment returns and select financial metrics presented in Table 1-5 were calculated based on three gold price scenarios.

Table 1-5: Summary of Economic Analysis Results (PEA Base Case and Two Alternate Cases)

Base Case – Gold Price at US$2,600	**NPV$^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$326.1M	82.5%	1.0 year	2.5
After-Tax	$215.8M	60.3%	1.2 year	1.7
Alternate Case 1 – Gold Price at US$3,400	**NPV$^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$681.2M	148.9%	0.6 year	5.3
After-Tax	$442.0M	107.9%	0.8 year	3.4
Alternate Case 2 – Gold Price at US$4,200	**NPV$^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$1,036M	212.1%	0.5 year	8.1
After-Tax	$668.3M	153.2%	0.6 year	5.2

Table 1-6 shows the cash flow and the key economic parameters.

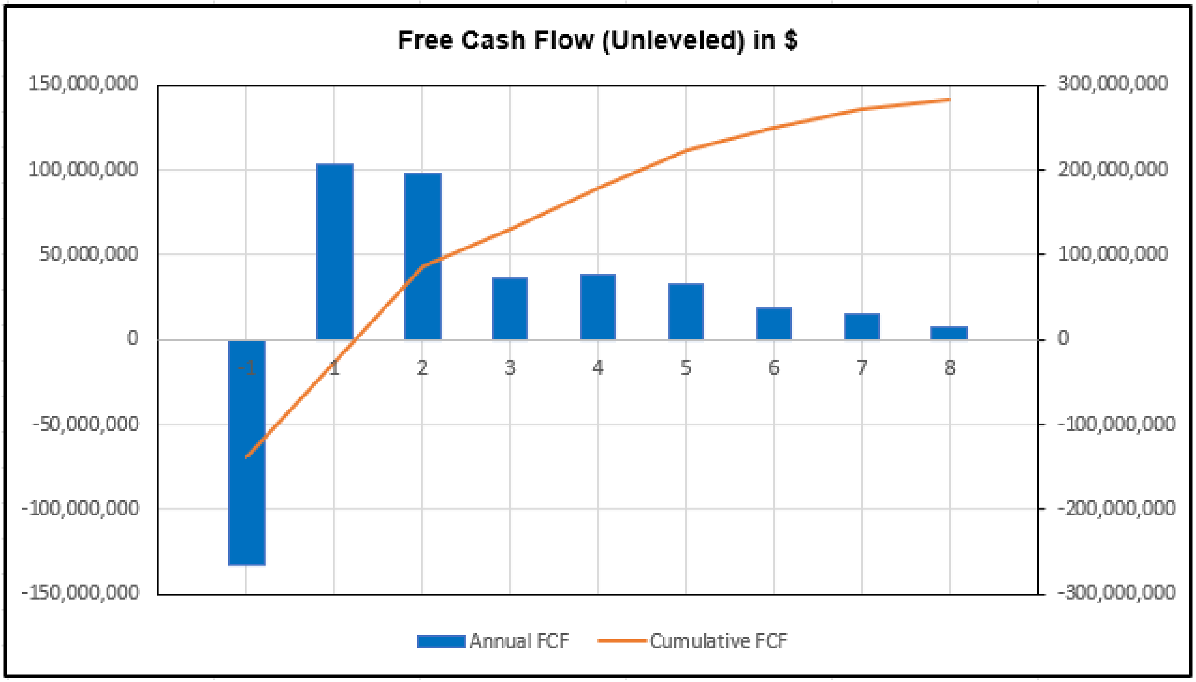
Table 1-6: Economic Analysis Summary (2025 PEA Base Case)

Description	Value	Units
Gold Price	2,600	US$/oz
Canadian Dollar to US Dollar Exchange Rate	0.72	C$:US$
Average Processing Throughput	900	tpd
Mine Life	7	year
Milled Tonnage	2.19	Mt

Description	Value	Units
Average LOM Gold Head Grade	6.86	g/t
Contained Gold	483,000	oz
Gold Recovery	94.7	%
Payable Gold, LOM	457,600	oz
Average Annual Gold Production (LOM)	65,400	oz
Average Annual Gold Production (Year 1 to 4)	77,300	oz
Total Operating Cost	185.40	$/t processed
UG Mining Cost	118.10	$/t processed
OP Mining Cost	6.95	$/t mined
Processing Cost	17.96	$/t processed
G&A Cost	31.23	$/t processed
Surface Services Cost, including Water Management	16.76	$/t processed
Initial Capital Cost	128.6	$ million
LOM Sustaining Capital Cost	76.7	$ million
LOM AISC	1,452	US$/oz Au
Before-Tax IRR	82.5	%
Before-Tax NPV	326.1	$ million, 5% discount rate
Before-Tax Undiscounted LOM net free cash flow	419.1	$ million
Before-Tax Payback period	1.0	year
After-Tax IRR	60.3	%
After-Tax NPV	215.8	$ million, 5% discount rate
After-Tax Undiscounted LOM net free cash flow	283.5	$ million
After-Tax Payback period	1.2	year

The project's post-tax FCF summary is shown in Figure 1-7.

Figure 1-7: Post-Tax Annual and Cumulative FCFs



Federal, Provincial, and BC Mineral Tax payables based on the PEA financial model are calculated at three different gold prices, and the resultant cash flow summary is shown in Table 1-7. The BC Mineral Tax (Provincial Resource Tax) is deductible from Federal and Provincial taxes payable.

Table 1-7: Project Cash Flow Summary

Gold Price (US$/oz)	2,600	3,400	4,200
Net Cash Flow (Before-tax, $ millions)	419.1M	857.8M	1,296M
Net Cash Flow (After-tax, $ millions)	283.5M	561.9M	840.5M
NPV @5% (Before-tax, $ millions)	326.1M	681.2M	1,036M
NPV @5% (After-tax, $ millions)	215.8M	442.0M	668.3M

1.16 Conclusions and Recommendations

The Scottie Gold Mine Project is considered to be technically and economically viable based on the results of the work presented in this Technical Report. It is recommended to advance the Project to the next stage. Section 26.0 outlines detailed recommendations for the Scottie Gold Mine Project.

2.0 INTRODUCTION

2.1 Overview

The Scottie Gold Mine Property is currently 100% owned by Scottie Resources Corp. (Scottie). Scottie commissioned Tetra Tech Canada Inc. (Tetra Tech) to complete this Technical Report, in accordance with NI 43-101 Standards of Disclosure for Mineral Projects.

The geology sections of this Technical Report were completed by Moose Mountain Technical Services (MMTS), including project description and location, accessibility, history, geological setting, deposit types, exploration, drilling, Mineral Resource estimate and adjacent properties.

Description and location of the Scottie Gold Mine Property are presented in Section 4.0 of this Technical Report.

2.2 Terms of Reference

This Technical Report was prepared for Scottie to summarize the results of the 2025 Pre-Economic Assessment (PEA).

The Report includes the maiden Mineral Resource Estimate of the Blueberry and the Scottie Mine deposits that were previously disclosed in the NI43-101 Technical Report titled "2025 Maiden Mineral Resource Estimate for the Scottie Mine Project" with the effective date of ebr uary 2, 2025, which was filed on SEDAR on June 23, 2025. The Qualified Persons (QPs) that authored the Technical Report are independent of Scottie and the Property.

2.2.1 2025 PEA Press Release

The results of the 2025 PE were disclosed in Scottie's press release dated ctober 28, 2025. This report is filed in support of the disclosure of the 2025 PEA results.

2.3 Sources of Information

The key information sources for this report were:

- Documents referenced in Section 3.0 (Reliance on Other Experts)
- Documents referenced in Section 27.0 (References)
- Additional information provided by Scottie personnel where required

2.4 Effective Dates

The overall effective date of the PEA Report is October 28, 2025.

The effective date of the Mineral Resource estimate is February 2, 2025.

2.5 Qualified Persons

The names of the Qualified Persons (QPs) of this report and their QP certificates are included in Section 28.0.

2.6 Personal Inspections

The following QPs conducted a site visit of the Property:

- Sue Bird, P.Eng. of MMTS, visited the site on September 7, 2024. During the site visit collar locations at Blueberry, Bend, Gulley and the Scottie Mine deposits were validated. The core storage site in Stewart was visited, with core from each deposit examined for mineralization and seven samples for re-assay obtained for validation of previous assay results.
- Hassan Ghaffari, P.Eng. of Tetra Tech, visited the site on July 29, 2025, and conducted a general project site overview in the proposed infrastructure areas.
- Damian Gregory, P.Eng. of Datamine Canada Inc. (Snowden Optiro), visited the Property on July 29, 2025, and conducted a general project site overview in the proposed open pit, rock storage and underground mine portal areas.
- Jianhui (John) Huang , Ph.D., P. Eng., of Tetra Tech, visited Sepro Metallurgical Laboratory on August 27, 2025, and inspected the sample tested and laboratory facility and witnessed the Dense Media Separation (DMS) test sample preparation.

3.0 RELIANCE ON OTHER EXPERTS

The QPs of this report state that they are qualified persons for those areas as identified in the "Certificate of Qualified Person". The QP has relied, and believe there is a reasonable basis for this reliance, upon the following other expert reports, which provided information regarding mineral rights, surface rights, and environmental status in sections of this report as noted below.

3.1 Mineral Tenure and Surface Rights

The Geology QP, Sue Bird, P.Eng., has not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area, or underlying property agreements. The QP has fully relied upon, and disclaims responsibility for, information supplied by Scottie Resources Corp., experts and experts retained by Scottie for this information through the following documents:

- Letter from Scottie Resources Corp. on claims and Royalties (Mumford, 2025) This title information is used in Section 4.0 of the Report.

3.2 Royalties and Incumbrances

The Marketing Studies and Contracts QP and the Economic Analysis QP, Jianhui (John) Huang, PhD, P.Eng. and Hassan Ghaffari, MASc,, P.Eng., respectively, have not reviewed the royalty agreements nor independently verified the legal status of the royalties and other potential incumbrances. The royalty structure on the project changed since the 2021 43-101 report. Franco-Nevada Corporation now holds a 2.0% gross production royalty on all claims on the project. This is the only royalty on the Scottie Gold Mine Project, all historic royalties on the Scottie Project have been eliminated.

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Scottie Resources Corp. for this information through the following documents:

- Letter from Scottie Resources Corp. on claims and Royalties (Mumford, 2025). This title information is used in Section 4.0, 19.0 and 22.4 of the Report.

3.3 Marketing Studies and Contracts

Dr. Jianhui (John) Huang, Ph.D., P.Eng. relied on the Letter from Scottie Resources Corp. on marketing terms (Mumford, 2025) for matters relating to the smelting terms, refining terms, saleability, and sales terms for the sellable products – the basis of which is a long term agreement with Ocean Partners to purchase the products from the Scottie Gold Mine. These terms are summarized and applied in Section 22.2.

3.4 Taxes

Mr. Hassan Ghaffari, P. Eng. has relied upon the taxation expert retained by Scottie for the taxation information applied in Section 22.5 of the Report.

3.5 Environmental Studies, Permitting, and Social Or Community Impact

Mr. Hassan Ghaffari, P. Eng. has relied upon the environmental studies, permitting, and social or community impact expert retained by Scottie for the information in Section 20.0 of the Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project is located in northern BC as illustrated in Figure 4-1. The property consists of 21 contiguous mineral claims covering an area of 8,840 hectares and an additional 14 Crown-granted claims for a total area of 9,053 hectares as illustrated in Figure 4-2. The Property is located 35 km north-northwest of Stewart, British Columbia, and is centered at 56°11′N, 130°07′W; 433,000 E, 6,232,000 N (NAD 83).

The property is found within the Skeena Mining Division and the claim boundaries were obtained from government claim maps. The crown granted claims, and mineral tenures are entirely owned by Scottie Resources. The claims/crown grants are subject to a 2% Gross Production Royalty held by Franco-Nevada. No other royalty or encumbrance exists on the claims.

An all-season camp facility exists at the Scottie Gold Mine Project site. The camp is equipped with diesel generators, a satellite communication link, tent structures on wooden floors, and several wood-framed buildings. The Scottie Resources offices and additional facilities and core storage are located in the town of Stewart.

A full Claims List can be found in Table 4-1.

Figure 4-1: Location of Scottie Gold Mine Project

(Source: Scottie Resources Corp., 2025)

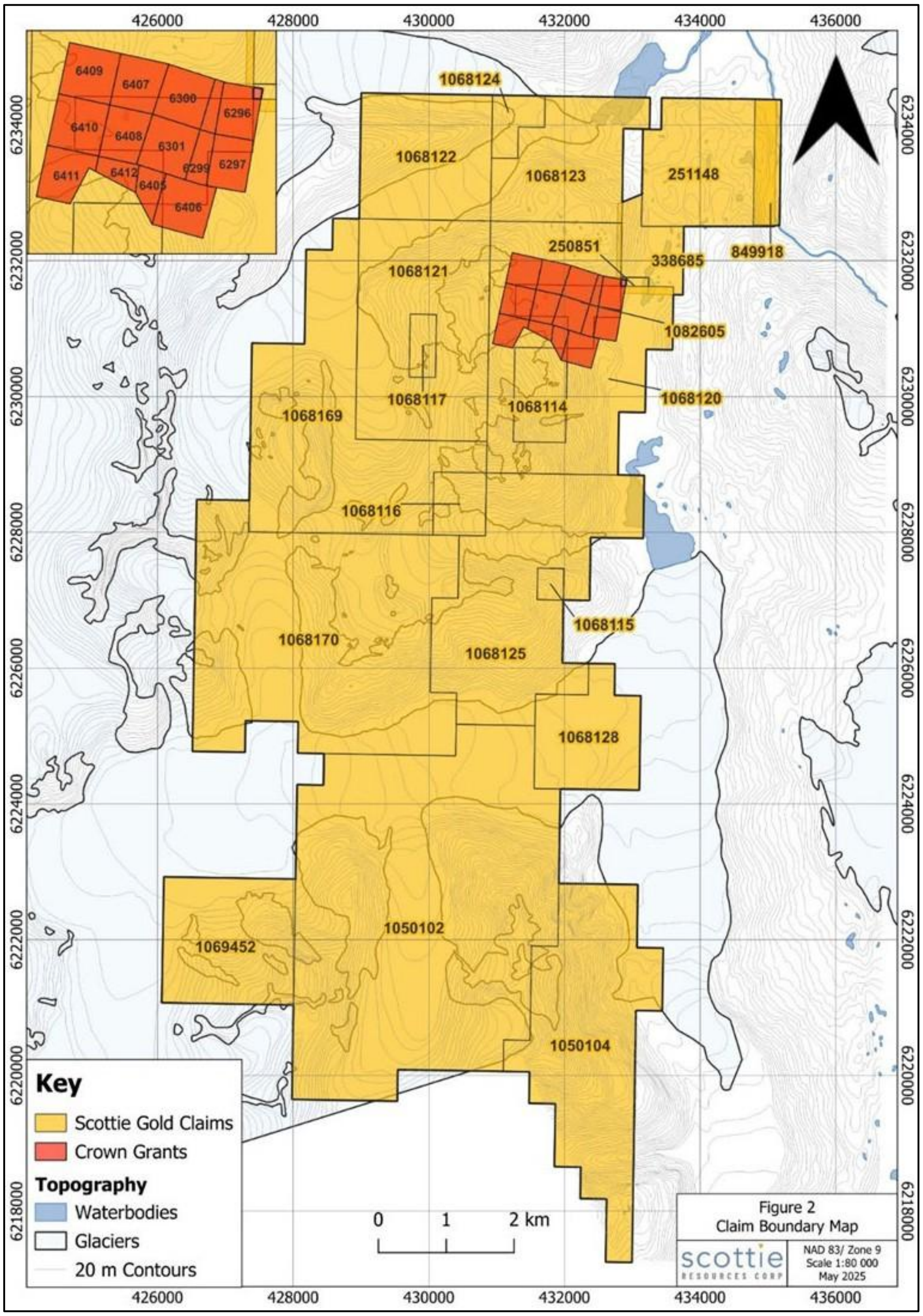


Figure 4-2: Scottie Resources Claims Map of the Scottie Gold Mine Project

(Source: Scottie Resources Corp., 2025)

Table 4-1: List of Claims

CROWN GRANTS:

DISTRICT LOT	MTA CGMC	CLAIM NAME	LOT_STATUS	ISSUE DATE	RNHCT RS	Percent Ownership
6405	6521	SUMMIT LAKE NO. 7	CROWN GRANTED	20-Dec-56	7.8	100
6407	6522	PRINCE NO. 1	CROWN GRANTED	20-Dec-56	18.5	100
6410	6523	PRINCE NO. 5	CROWN GRANTED	20-Dec-56	20.9	100
6411	6524	PRINCE NO. 6	CROWN GRANTED	20-Dec-56	17.3	100
6300	6539	SUMMIT LAKE NO. 5	CROWN GRANTED	20-Dec-56	19.7	100
6301	6540	SUMMIT LAKE NO. 6	CROWN GRANTED	20-Dec-56	20.1	100
6299	6544	SUMMIT LAKE NO. 4	CROWN GRANTED	20-Dec-56	7.5	100
6297	6562	SUMMIT LAKE NO. 2	CROWN GRANTED	20-Dec-56	15.7	100
6409	8886	PRINCE NO. 4	CROWN GRANTED	20-Dec-56	20.9	100
6296	9869	SUMMIT LAKE NO. 1	CROWN GRANTED	20-Dec-56	15.7	100
6298	9870	SUMMIT LAKE NO. 3	CROWN GRANTED	20-Dec-56	4.9	100
6406	9873	SUMMIT LAKE NO. 8	CROWN GRANTED	20-Dec-56	20.6	100
6408	9874	PRINCE NO. 2	CROWN GRANTED	20-Dec-56	16.5	100
6412	14078	PRINCE FRACTION	CROWN GRANTED	20-Dec-56	6.9	100

MINERAL CLAIMS:

CLAIM #	CLAIM NAME	ISSUE DATE	GDTDT	RNHCTRS	CLIENT #	OWNER NAME	Percent Ownership
250851	SCOT #4	19800213	20340602	150	289731	SCOTTIE RESOURCES CORP.	100
251148	BOW 1	19840125	20340602	400	289731	SCOTTIE RESOURCES CORP.	100
338685	SUM #1	19950804	20340602	150	289731	SCOTTIE RESOURCES CORP.	100
849918		20110327	20340602	71.9193	289731	SCOTTIE RESOURCES CORP.	100
1050102	STOCK2	20170217	20340602	1802.753	289731	SCOTTIE RESOURCES CORP.	100
1050104	STOCK3	20170217	20340602	703.359	289731	SCOTTIE RESOURCES CORP.	100
1068114		20190425	20340602	107.9719	289731	SCOTTIE RESOURCES CORP.	100
1068115		20190425	20340602	18.007	289731	SCOTTIE RESOURCES CORP.	100
1068116		20190425	20340602	72.0086	289731	SCOTTIE RESOURCES CORP.	100
1068117		20190425	20340602	35.9848	289731	SCOTTIE RESOURCES CORP.	100
1068120		20190425	20340602	629.7163	289731	SCOTTIE RESOURCES CORP.	100
1068121		20190425	20340602	593.7148	289731	SCOTTIE RESOURCES CORP.	100
1068122		20190425	20340602	359.6141	289731	SCOTTIE RESOURCES CORP.	100
1068123		20190425	20340602	323.6539	289731	SCOTTIE RESOURCES CORP.	100
1068124		20190425	20340602	53.9353	289731	SCOTTIE RESOURCES CORP.	100
1068125	FIN	20190425	20340602	720.2969	289731	SCOTTIE RESOURCES CORP.	100
1068128		20190425	20340602	234.2098	289731	SCOTTIE RESOURCES CORP.	100
1068169	BINER	20190427	20340602	755.8857	289731	SCOTTIE RESOURCES CORP.	100
1068170	ATC	20190427	20340602	1206.695	289731	SCOTTIE RESOURCES CORP.	100
1069452	ROPE BURN	20190703	20340602	360.5854	289731	SCOTTIE RESOURCES CORP.	100

CLAIM #	CLAIM NAME	ISSUE DATE	GDTDT	RNHCTRS	CLIENT #	OWNER NAME	Percent Ownership
1072155	QU59	20191028	20340602	89.9488	289731	SCOTTIE RESOURCES CORP.	100

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

5.1 Accessibility

The well-maintained Granduc Mine Road passes through the claim boundaries, which provides access to the northeastern areas of the Property. Helicopter support is used to provide access to the remaining portion of the property. Scottie seasonally sets up a summer airbase at site, supported through Stewart.

5.2 Climate

The climate is classified as humid continental, with an average annual temperature of 6.1°C, and an average yearly precipitation 1866 mm. Field work on the Property is typically carried out from late June to October; however, the Granduc Mine Road could be kept open throughout the year with the use of an adequate avalanche control program and a snow removal team.

5.3 Local Resources and Infrastructure

The closest power source is the Brucejack Mine Transmission Line (BMTL) located less than one kilometre from the eastern claim boundary. The BMTL is owned by Newmont Corporation while the power is provided by BC Hydro, the public utility.

The town of Stewart, BC, has a population of approximately 500, and is host to an ice-free deep-water port, a paved airstrip, as well as several stores that supply basic amenities. The city of Terrace and town of Smithers are both approximately four hours away by vehicle from Stewart. Both provide access to commercial airports along with most other services required to support mineral exploration and mining projects.

Figure 5-1 is a photo illustrating the camp site during the 2024 drill season.

Figure 5-1: Camp Site Currently Used at Site (Seasonally Borrowed from another Exploration Company)

(Source: MMTS, 2025)

5.4 Physiography

The Property is located within the Boundary Ranges of the Coast Mountains. The elevation on the property ranges from a low of 700 m above sea level (asl) to a high of 2,126 m (asl), found at the peak of Summit Mountain. The eastern portion of the Property is host to slopes with lower elevation which are sparsely vegetated with spruce trees, alder trees, and blueberry bushes. Most of the property is in the alpine with much of the western portion being heavily glaciated. Figure 5-2 shows a photo of the Scottie Gold Mine Property illustrating the general physiography.

Figure 5-2: Photo of Scottie Mine Area
(Source: Scottie Resources Corp., 2025)

6.0 HISTORY

The following section is provided by excerpts from Horvat and Thorogood 2024, and Gunning and Visagie, 2006.

The first exploration undertaken in the Stewart area was in 1898 by prospectors exploring the area on their way to the Klondike gold rush. Some of the earliest staked claims in the area included the Grizzly Bear property near Bitter Creek in 1899, the American Boy property on American Creek in 1900, and Glacier Creek in 1903.

Currently the Scottie Gold Mine Property encompasses several prospects, each of which with their own separate exploration history which has been compiled from various authors including Tribe et al. (1983), Tribe (1985), Dick (1987), Visagie and Varas (1991), Gunning and Visagie (2006), Kruchkowski (2017), Voordouw and Carr (2019), and Nuttall et al. (2021). The work completed up until 2018 on the Scottie Gold Mine, Blueberry and Bend, and the surrounding prospects are independently discussed below.

6.1 Scottie Gold Mine History

This section is mainly related to the Scottie Gold Mine and adjacent prospects, including the C and D Zones, and is compiled from reports by Tribe et al. (1983), Tribe (1985), Dick (1987), Visagie and Varas (1991), Kruchkowski (2017), Gunning and Visagie (2006), and Voordouw and Carr (2019).

The initial discovery of gold-bearing veins in the Scottie Gold Mine area was in 1928, by Ted Morris and s sociates of Stewart, BC, who then staked the main surface showings under the name "Salmon Gold". The property was optioned to Premier Gold Mining Company ("Premier") in 1931, with surface sampling and trenching operations revealing ore-grade mineralization in two zones along-strike lengths of 85 to 350 ft. Between 1931 and 1934, Premier completed 10 diamond drillholes with six indicating a downward extension of the veins. The option lapsed and the property returned to the newly incorporated Salmon Gold Mines Ltd., who subsequently optioned it to Consolidated Mining and Smelting Company of Canada ("Cominco") in 1934.

Cominco drilled several surface drillholes in 1934, with encouraging results. This was followed up between 1935 and 1938 with the development of a hand-steeled adit and 5,000' of cross-cutting and drifting at the 3,600' elevation of the property. The underground development revealed 210' of strike length of the main vein, presumed to be M Zone, with average widths of 2.4' and grades of 0.357 oz t u. During this period, 3,000' of underground diamond drilling showed that one zone had potential for at least 1,000' of vertical extent.

Cominco dropped its option in 1939. The property then sat idle until 1946. Between 1946-1948, Morris Summit Gold Mines Ltd. ("Morris Summit") completed 17,000' of diamond drilling, an additional 4,000' of lateral development work and raise development from a new portal developed at the 3,000' level. These efforts helped identify four mineralized shoots including the McLeod East Zone (now part of the M Zone), which was intersected with a spur crosscut directed toward an old Cominco Drillhole.

During the underground development by Morris Summit, work was completed by Letal Exploration at the C and D Zones in 1946-1947, known then as Scottie South and Scottie North respectively, following the discovery of shear-hosted mineralization. The C and D Zones are located above the western shore of Summit Lake, approximately 1.5 km northeast of the 3600' portal. Work consisted of 1,300' of trenching, rock sampling, and 2,730' of diamond drilling from 13 holes. Mapping and sampling determined that the most significant gold-bearing mineralization is where north trending fractures intersect northeast trending fractures. The best intercept of the diamond drill program was 0.454 oz/t u over 6.5' at the C Zone (Seraphim, 1947).

Morris Summit was unable to develop the other three mineralized shoots due to a lack of funding and once again the property sat idle until 1952. In 1952, a joint venture between Newmont and Granby Mining & Smelting Company gained control of Morris Summit. In 1955, they re-sampled the historical workings and diamond drillholes to substantiate the historical findings then followed this up in 1956 with surface prospecting and geophysics, which outlined several more gold-bearing veins. However, no follow-up work was conducted on any of these veins and the property remained idle until 1978 when the controlling interest of Morris Summit was acquired by D. A. McLeod and Associates of Vancouver ("McLeod Group").

The McLeod Group added the C and D Zone prospects to the original Morris Summit claim block and formed Scottie Gold Mines Ltd. to develop the property into commercial production. With the new land package, the McLeod Group developed an access road through to the 3,000' level adit in 1978 and completed 3,058' of diamond drilling at the M Zone between 1978 and 1979. d ditional development was also conducted on the N Zone and on the McLeod West veins. A feasibility study, completed in 1980 and based on a gold price of $660/ounce, recommended placing the property into production at 200 tons/day. Reserves in all categories were calculated to be 226,287 short tons averaging 0.743 oz/short ton Au, undiluted (this a historical estimate and these reserves are not compliant with current NI-43- 101 standards).

The Scottie Gold Mine was put into production on October 1, 1981, and continued for about 3.5 years until February 18, 1985, when high interest rates and a low price of gold ($300/oz) forced the mine into closure. Total production amounted to 95,426 oz Au from 201,462 short tons milled, for an average recovered grade of 0.474 oz/short ton Au or 16.25 g/t Au. Underground mining was done by shrinkage stoping varying from 3.5' to 30' in width with track haulage to an underground mill. Mining varied from 135 to 185 tons/day throughout its production. Recovered grades averaged 0.51 oz/short ton Au for the first two years but dropped to 0.41 oz/short ton Au in 1984. Lowered grades were attributed partly to narrower vein widths, with attendant higher dilution, but was also due to the mining of lower grade material to maintain tonnage. Most of the production between 1981 and 1985 was from the M Zone, with lesser amounts produced from the N and O zones.

s part of the mining process, a total of 45,188.8' (13,777.1 m) was drilled in 201 underground drillholes from 1981 to 1985. Most of the drilling was focused on the evaluation (and subsequent mining) of the M, N and O zones. Limited drilling of the M Zone was conducted beyond the mined-out extent and resulted in the identification of several significant intercepts such as 11.3' at 0.642 oz ton u, 0.8' at 5.518 oz ton u and 5.5' at 0.572 oz ton u (all true widths). In addition to this, three holes were drilled to test 100' of strike length along the L Zone with all three holes returning narrow intersections with anomalous gold, including 3.0' at 2.036 oz ton u and 1.0' at 1.576 oz ton u (true widths).

Gunning and Visagie, 2006 report that at shutdown company personnel prepared a shutdown report that outlined the potential of the property in the vicinity of the main workings. The results of this report are summarized below (Table 6-1).

Table 6-1: Historical Reserves at Scottie Gold Mine, 1985 Shut Down

Category	Description	Tons (short tons)	Au Grade (opt)	Ounces
Proven	Material that is exposed within the vein openings and within a known vein. This category is projected 25 feet from a mine opening along the vein.	29,265.62	0.54	16,013.68
Probable	Extensions of ore away from proven ore but within the known vein for a distance of 25 to 50 feet. Drillhole intersections within the known vein and around that intersection for a distance of 25 feet within the vein.	74,893.84	0.54	40,986.20
Possible	Ore projected 50 to 70 feet from a mine opening and within the vein. Ore projected 25 to 50 feet from a Drillhole and within a known vein.	28,146.93	0.61	17,332.09
Total		132,306.00	0.56	74,331.00

These resources estimates do not follow the required disclosure for reserves and resources as outlined in National Instrument 43-101 as they were prepared in the 1980’s prior to the implementation of the instrument. The historic estimates generated by company personnel have not been redefined to conform to the CIM approved standards as required in NI 43-101. The resource estimates cannot be verified. The QP has made no effort to refute or confirm these estimates. They can only be described as historical estimates.

The tonnage estimate was completed using the chip sample data from the mine workings or using vein widths as determined by drilling. A minimum mining width of 3.5 feet was used. The cut-off grade was 0.30 opt Au over a minimum mining width of 3.5 feet. High-grade assays were cut according to the following format:

- Isolated highs are cut to 2.00 opt Au.
- Three or more adjacent high-grade assays across a vein have the two outside assays cut to 2 opt Au with the remaining left uncut.

Assays were cut before dilution was considered. The tonnage factor used was 10.3 cubic feet per ton.

Surface exploration programs were also conducted while the mine was in production, particularly during the summers of 1983 and 1984. This work included approximately 150 miles (~240 km) of airborne geophysical surveys (VLF-EM, magnetics) followed up with 16 miles (25 km) of ground-based geophysics. The airborne geophysical survey covered most of the current claim package and identified numerous EM anomalies. Ground-based geophysics identified 11 mag anomalies and 10 EM conductors in the C Zone area. Property-wide mapping and sampling was completed at 1":200' scale while underground mapping of the immediate mine surface was completed at 1":40' scale. Surface diamond drilling during this period totaled 3,994' in 15 drillholes targeting the C, D, O and S zones.

Results from the C Zone in 1983 include intercepts of 2.7' at 2.263 oz ton u and 6.5' at 0.454 oz ton u. In 1984, follow up drilling at the C Zone intersected 4.5' at 0.16 oz ton u and 4' at 0.316 oz ton Au. At the S Zone, now interpreted to be the surface expression of the M Zone, results included a 10' true width intercept averaging 0.492 oz ton u, while the Zone results included a 5' intercept averaging 1.239 oz/ton Au. Two kilometres west of the Scottie Gold Mine at the Back Grid, now known as the Domino Zone, rock sampling of massive sulphide float returned strongly anomalous gold values and a geophysical survey completed over a glacier identified EM conductors with coincident low mag readings.

The mine was placed into receivership by the Royal Bank of Canada in February 1985, with a re-organization of Scottie Gold Mines resulting in the formation of a new company – Royal Scot.

Exploration on the property resumed in 1987, with Royal Scot completing 18 underground drillholes for 5,214'. These holes were drilled to test the M, N and L zones along-strike and downdip. The holes showed that the M Zone continued below the existing workings and to the west of the mined-out area. Drilling on one section returned over 230' of highly anomalous gold values. Results include intercepts of 6.3' at 2.313 oz t u and 4.5' at 1.345 oz t u (true widths). The underground holes were drilled from the 3,000' level and showed that mineralization continued down to the 2,800' level. Two holes drilled approximately 90' apart and 80' below the N Zone intersected 0.281 oz t u and 0.249 oz t u over true widths of 5.38' and 5.29' respectively.

Exploration work in 1989 comprised a review of the historical data coupled with a two-day prospecting and mapping program. Several new showings were identified with select grab samples assaying up to 1.219 oz/t Au and 2.45 oz/t Ag.

In 1990, Royal Scot completed a more comprehensive evaluation of the project that included both surface and underground drilling, mapping, sampling, prospecting, and geophysical surveying of select areas. total of 13,940' of geophysical surveys (VL -EM, magnetometer) were completed on grids located on the C and D zones while rock chip sampling and mapping was completed in various zones throughout the project area. Rock sampling in the C Zone returned up to 3.715 oz/t Au and up to 1.572 oz/t Au in the D Zone. Four underground drillholes totaling 1,791.80' tested the downdip and along strike extension of the M Zone with two of these holes returning encouraging values, such as 5.6' averaging 0.564 oz t u (true width). Two surface holes were also drilled to test the C and E Zones. t the C Zone, a 7.5' intersection averaged 0.164 oz t u while at the E Zone a 1' intersection returned 0.042 oz/t Au. The Sulphide Zone at the Back Grid was discovered, mapped, and sampled with grab sample assays up to 1.11 oz/t Au and chip samples up to 0.81 oz/t Au. A drillhole was established to follow up on these results but was abandoned after 10 m due to poor ground and adverse weather.

Royal Scot merged with Tenajon in 1991, and adopted the name of Tenajon Resources Corp. The new company conducted limited soil, silt, and water sampling on the west side of Summit Lake in 1994 and partial site reclamation in 1995. In 1996 Arkaroola Resources Ltd. (“ r karoola”) entered into an agreement to purchase the property from Tenajon and conducted a limited soil and rock chip sampling program in 1997, approximately 200 m to the west of the area sampled in 1994. However, the property was returned to Tenajon in 1998 as Arkaroola was unable to maintain the schedule of payments. Tenajon conducted additional site reclamation in 1998, but the project remained idle once again until 2004.

A data review in 2003, identified several drill targets near the existing mine workings and elsewhere on the property, leading to a 14-hole underground drill program and a modest prospecting program in 2004. The drill program tested the M, N and L zones. An additional 19 drillholes for 2,028 m were completed in 2005 that successfully expanded these same three zones along strike and down-dip. The M Zone returned several high-grade intercepts, and the zone was interpreted to be open along strike to the west. An NI-43-101 report was prepared on the project following these results.

Drilling continued in 2006, with a 31-hole drill program designed to test several zones, with 3,650 m of underground drilling resulting in the discovery of the “R Zone” 137 m south of the M Zone, now interpreted to be an extension of the L Zone.

In addition to work completed by Tenajon, Seeker Resources Ltd. completed a reconnaissance sampling program in 2006 within the first 3 km of the Granduc Tunnel, a majority of which underlies the current claims to the north of the Scottie Gold Mine. Quartz-sulphide fissure veins located between 2290-2325 m were sampled and produced gold values of 4.82 g/t, 23.50 g/t, and 35.70 g/t (Kikauka, 2007).

In 2008, the project was purchased from Tenajon by Jayden Resources, who subsequently sold it to Red Eye in 2009. From 2010 to 2014 Red Eye embarked on a series of limited work programs, including data compilation, site investigation, underground sampling, environmental studies, government discussions, an evaluation of the condition of on-site milling and crushing equipment and pre-feasibility studies. In 2014 Red Eye sold an 80% stake in the Scottie Gold Mine Property to Rotation Minerals Ltd.

In 2015, Eilat Resources Ltd. carried out rock and soil sampling on the now dry lakebed of Summit Lake. Rock sampling returned 7.1 g/t and 19.05 g/t Au at a new showing named Yom Kippur. In total, 30 rock, 26 soil, and 9 tailing samples were collected. Five rock samples taken just west of Scottie Gold Mine Camp Portal all assayed above 1 g/t and up to 6.74 g/t Au. Soil samples collected in the Yom Kippur zone produced values of up to 1.42 g/t Au (Kikauka 2016).

In 2016, Rotation embarked on a rock sampling and drilling program on the project. A total of 162 rock samples were collected from four main areas: 3,600’ portal, the C and D zones, 6 oz. Zone and Dave Zone. The 3,600’ portal area was considered the most prospective as it features several high-grade gold- silver veins, including one that is 0.4-2.0 m wide and has at least 100 m of strike length. A grab sample from this vein returned 151 g/t Au, 106 g/t Ag, 0.1% Cu and 0.8% Zn. In the C Zone area, rock sampling produced values of up to 447.95 g/t Au. One float sample in the 6 oz. Zone returned 186.6 g/t Au with in-situ samples assaying up to 11.21 g/t Au. The diamond drilling program comprised 2,648.78 m in 21 drillholes from 5 pads, 1,935.36 m of which was drilled from surface at the C and D zones.

The remaining 713.42 m was drilled at the Dave Zone to test areas of silicification coupled with closely spaced pyrite veins carrying minor pyrrhotite. The best drill results were returned from the C Zone, including 1.13 m of 31.54 g/t Au and 4.81 m of 5.18 g/t Au (Kruchowski, 2017).

In December 2017, Rotation completed the purchase of a 100% interest in the Scottie Gold Mine Property and thus became the sole proprietor of the historical mine (Scottie Resources, 2017).

In 2018, Rotation Minerals completed prospecting, relogging and resampling of historical drill core, auger sampling tailing piles, and surface reclamation work. In total, 45 rock samples were collected while prospecting and mapping the 2-kilometre-long access tunnel between camp and 3,000’ level. Relogging of 25 drillholes for 3,113.7 m from the 700-series historical drill core was conducted. 115 samples were collected, 52 of which were re-samples of previously sampled intervals. Results for resampled intervals returned 1-7% higher gold values than historical results. No significant results were returned from the previously unsampled intervals. Auger sampling of the tailings pond outside of C Portal in the Summit Lake basin was completed from 14 holes with an average of 5-6 samples and maximum depth of 3.1 m, for a total of 74 samples. Gold assays ranged from 0.2-6.2 g/t Au for all 74 samples and averaged 2.1 g/t Au. Portals A, C1, C2, and D were reframed and sealed.

6.2 Blueberry and Bend History

The claims comprising the Bend and Blueberry showings were originally within the Tide Lake mineral reserve established December 4th, 1967, by order-in-council at the request of Newmont Mining Corporation to protect the mill, tailings disposal area, airstrip and planned townsite for workers of the Granduc Mine. In 1983, Esso Resources Canada Ltd. (“Esso”) carried out prospecting within the mineral reserve, locating a massive sulphide occurrence along the mine road that was named the Bend Vein showing. Initial assays from this pyrrhotite-rich shear vein returned 0.317 oz/t (8.99 g/t) Au across 17’ (5.2 m). grid was cut over an exposed 300 m strike length of the structure, which was then trenched, sampled, and mapped in detail. The discovery also motivated further mapping and sampling within the mineral reserve. Soil sampling and ground geophysical surveys (VLF-EM-16, time-domain IP, resistivity) were completed over the showing. Further work within the reserve identified a westerly continuation of the Bend Vein showing named the Cookhouse Zone, as well as locating three other sulphide-bearing structures that were later named the Blueberry Vein, Road showing, and Mill Vein. Recommendations included the suggestion that Esso argue for rescinding of the mineral reserve to the Provincial Government with the aim of obtaining mineral rights to the ground (Fraser et al., 1983).

In January of 1984, the Summit Group claims comprising the Bow 1, Wow 1, Wow 2, and Wow 4 claims was acquired through staking by a joint venture between Esso (50%) and Scottie Gold Mines Ltd. (50%). The claims were staked over ground released from the Tide Lake mineral reserve. Work completed by the joint venture group was focused on the Bow 1 claim and included: (1) mechanized stripping and trenching at the Bend Vein and Blueberry Vein showings, (2) 1:2,000 scale mapping at Bend and 1:500 scale mapping at Blueberry, (3) chip, grab, and channel sampling in trenches, (4) 330 soil samples collected over both showings, (5) geophysics, including HLEM, IP and ground-magnetics, and (6) 1,094 m of BQ diamond drilling, including five holes at Blueberry and 12 holes at Bend.

All 12 holes drilled at Bend intercepted the structure and the Bend Vein was intercepted over 60 m of strike length with an average vein width of 1.5 m. Assay results included an interval of 4.17 m averaging 82.2 g/t Au with 44.2 g/t Ag in hole SJV-07. The Blueberry Vein was exposed over 90 m; the five holes tested the vein from two sites 45 m apart. One hole intersected a 1.59 m interval averaging 26.56 g/t Au (SJV-11). Four additional drillholes tested other areas of the Bow 1 claim with no significant results reported. Esso recommended no follow up work (McGuigan and Wilson, 1985).

In 1989, data from previous years was reviewed, and a resource was calculated for the Bend and Blueberry showings (Petersen and Vulimiri,1989).

In 1989, Homestake purchased the assets of Esso Resources and in 1990 Homestake Mining (Canada) Ltd. entered into an agreement with Tenajon for exploration of the Bow 1 claim outside of the Bend Vein area. Soil sampling and limited mapping were undertaken. Results outlined a 150 x 600 m alteration zone located east of the Bend Vein hosting anomalous gold-in-soil values in association with a quartz vein stockwork (Unpublished).

In 1991, Tenajon completed ten short drillholes at the Bend Vein with the purpose of outlining a mineable reserve. The drilling traced the vein for more than 60 m to a depth of 40 m down dip. Follow-up drilling completed in 1992 did not outline any significant zones of interest.

In 2000, Homestake Mining assigned their interest in the Bow claim to Tenajon. In 2002, Tenajon completed soil sampling along the Bend Vein trend and completed minor reclamation work. Several samples returned >50 ppb Au and were collected along strike to the east of the Bend Vein.

In 2004, Tenajon completed limited prospecting and sampling at the Road Showing and an unnamed vein found running parallel to the Bend Vein.

In 2005, Tenajon completed 13 NQ diamond drillholes, for a total of 535.7 m, to test for strike and dip extensions. Results extended a high-grade portion of the Bend Vein along strike east and west, defining 110 m of strike and 50 m down dip extent. At Blueberry, two holes were drilled 8 m and 25 m north of SJV-11. Drillhole 05-13 intersected two mineralized zones; one vein hosted and the other disseminated in wall rock. Intercepts include 1.05 m of 0.475 oz/t Au and 1.61 m of 0.928 oz/t Au. Portions of this zone were intercepted in SJV-11 but not sampled. One hole (05-12) tested the Road Showing beneath a chip sample assaying 203 g/t Au across 0.6 m. Drilling intersected a true width intercept of 0.64 m averaging 4.04 g/t Au, 23 m down-dip from the surface sample.

In 2006, Tenajon completed diamond drilling, rock and soil sampling and ground-based geophysics. Five drillholes totaling 376.5 m were completed at Blueberry. Four holes tested the Grizzly Zone (located to the south of the Blueberry Vein), and one hole tested the Blueberry Vein, with Drillhole 06-3 (Grizzly Zone) returning 10.87 g/t Au over 1.02 m. Additional rock sampling from the Road Showing returned samples with up to 603 g/t Au and a sample taken immediately south of the Blueberry trench returned 108 g/t Au.

From 2007 to 2018, various work programs, including trenching, sampling, and drilling by Decade Resources, were completed at Blueberry, Bend, and the Stockwork Zone.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting

The following is summarized from Nelson et al. (2018) unless referenced separately and outlines the geology of the Stewart - McTagg - Snip map area.

The Stewart region is underlain by rocks of the Stikine volcanic island-arc terrane, situated within the Intermontane belt at the eastern edge of the Coast Plutonic Complex as illustrated in the stratigraphic column of Figure 7-1.

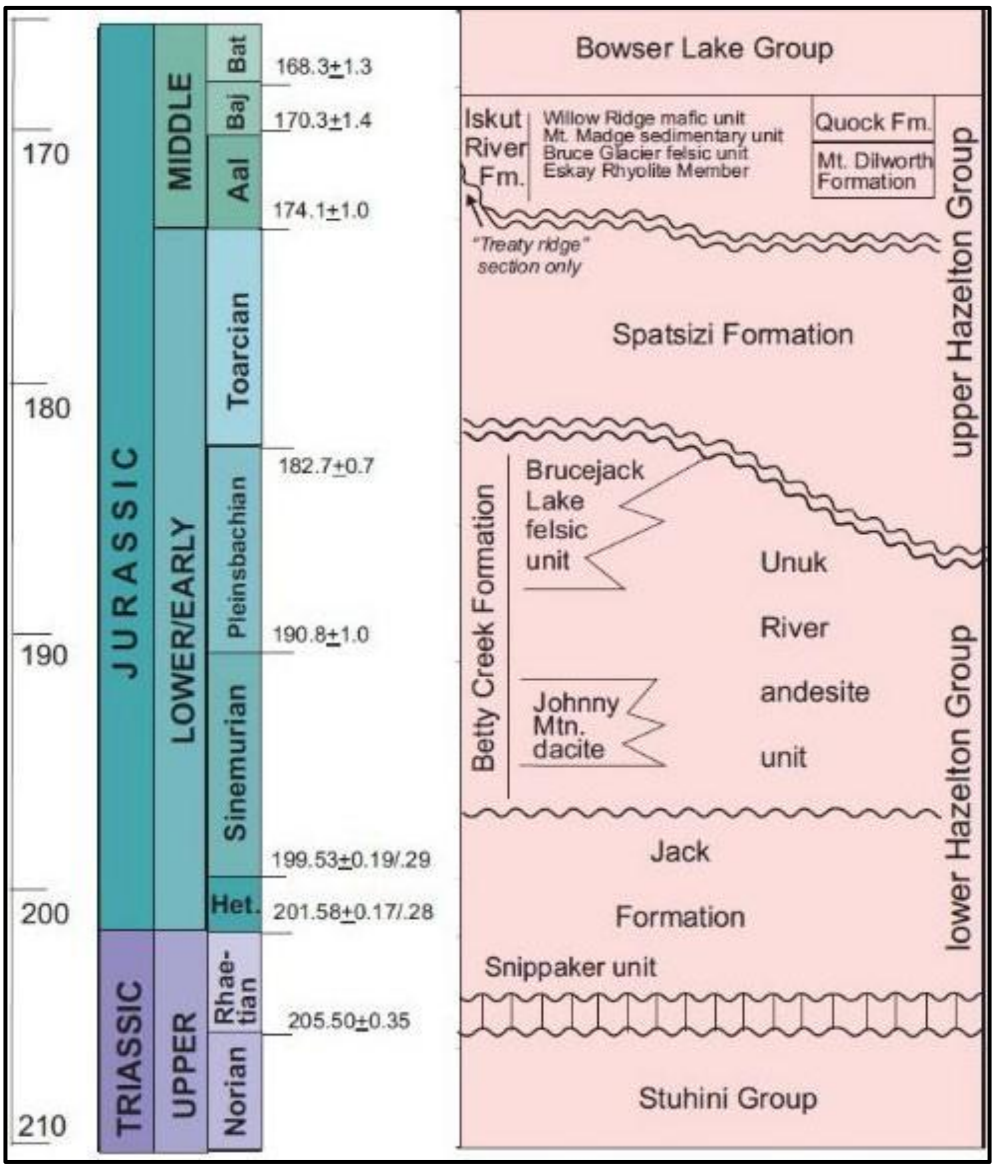


Figure 7-1: Stratigraphic Column of the Scottie Gold Mine Project District Geology

(Source: Scottie Resources Corp., 2025)

The Stikine represents a multi-stage arc terrane developed in an intra-oceanic setting isolated from the North American Margin and is composed of three uncomformably bounded successions: the Stikine Assemblage, the Stuhini Group, and the Hazelton Group.

Conformably overlying the Hazelton Group are siliciclastic cover rocks of the Bowser Lake Group that were deposited as the Stikine and inboard terranes accreted to the North American margin. Upper Triassic to Middle Jurassic volcano-sedimentary units of the Stuhini and Hazelton Groups, and Upper Jurassic to Lower Cretaceous sedimentary units of the Bowser Lake Group are present within or proximal to the Property. Intruding these groups are Jurassic to Eocene intrusions of the Texas Creek plutonic suite and Coast Plutonic Complex.

The Stuhini Group (Middle to Late Triassic) is regionally comprised of augite-phyric volcanic and volcaniclastic rocks, sedimentary rocks, and minor felsic volcanic rocks (Cutts et al., 2015). In the Stewart area, common lithologies consist of dark grey, laminated to thickly bedded, silty mudstone and fine- to medium-grained to locally coarse-grained sandstone. Less abundant lithologies include heterolithic pebble to cobble conglomerate, massive tuffaceous mudstone and thick-bedded sedimentary breccia and conglomerate.

A regional unconformity, marking a period of tectonic quiescence, forms the boundary between the Stuhini Group and Hazelton Group. The lower Hazelton Group, divided into the Jack and Betty Creek Formations, consists of volcanic and sedimentary rocks related to volcanism generated by the subduction of two opposing oceanic plates. At the base of the Hazelton Group, the Jack Formation (latest Triassic to early Jurassic) is discontinuously found in the region and is composed of conglomerate, sandstone, and siltstone with limey interbeds. This siliciclastic unit represents a significant break from Stuhini Group volcanic and volcaniclastic accumulation. Within this formation is the informal Snippaker Unit, a dull green greywacke with pebbles of hypabyssal diorite that increase in size up-section.

Overlying the Jack Formation, is the Betty Creek Formation (Lower Jurassic), consisting of the Unuk River andesite unit, Johnny Mountain dacite unit and Brucejack Lake felsic unit. The Unuk River andesite unit consists of subaerial and epiclastic deposits with a para-conformable to unconformable contact with the underlying Jack Formation. The Johnny Mountain dacite unit is a succession of bedded dacite lapilli tuff and breccia and in some areas unconformably overlies the Stuhini Group. The Brucejack Lake felsic unit overlies the Unuk River andesite unit and includes potassium feldspar-, plagioclase-, and hornblende- phyric flows, breccias, and bedded welded to non-welded felsic tuffs.

The upper Hazelton Group represents a period of arc demise, regional subsidence, and local development of the Eskay Rift. The Spatsizi Formation is the regional basal unit of the upper Hazelton and is comprised of a siliciclastic sequence of shale, siltstone, and sandstone with minor volcanic components.

The Iskut River Formation is a several kilometre thick succession and occupies the Eskay rift, a narrow, elongate north-trending belt extending from Kinaskan Lake in the north to Anyox in the south, running west of the Salmon River Valley and town of Stewart. It comprises a highly variable succession of mafic and felsic volcanic and sedimentary units that is subdivided into the Willow Ridge mafic unit, Bruce Glacier felsic unit, Eskay Rhyolite Member, and Mount Madge sedimentary unit.

Outside of the Eskay Rift, the Mount Dilworth Formation overlies the Spatsizi Formation, and is a felsic unit distinguished by its tabular geometry, regional extent, and lack of interfingering with mafic units.

The uppermost unit in the Hazelton Group is the Quock Formation and is informally known as the 'pyjama beds' unit. This aerially extensive layer is comprised of a 50-100 m thick sequence of thinly bedded, dark grey siliceous argillite with pale felsic tuff laminae.

Overlying the Hazelton Group is the Upper Jurassic to Middle Cretaceous Bowser Lake Group. Occupying a large area of the central Stikine, it is comprised of marine to non-marine sedimentary rocks, with the most widely occurring lithologies including sandstone and siltstone with lesser abundances of conglomerate.

Several late Triassic to Early Tertiary intrusions exist in the region. Late Triassic to Early Jurassic plutons are coeval and cogenetic with lower Hazelton volcanism and include the Tatogga suite, Texas Creek Suite, and Brucejack Lake Suite. The Texas Creek Suite, comprising of diorite, monzonite, and syenite porphyry intrusions, is the most widespread in the Stewart area and interpreted to be the subvolcanic equivalent of the Betty Creek Formation.

Early to Middle Eocene intrusions of the Hyder plutonic suite are found in the Stewart area and are associated with the northwest trending Early Cretaceous to Eocene Coast Plutonic Complex that lies on the western edge of the Stikine Terrane. In comparison to Early Jurassic intrusions, the calc-alkaline granite to tonalite to quartz monzonite plutons of the Hyder plutonic suite are biotite rich, more siliceous, and less altered. An extensive array of Tertiary granodiorite porphyry, aplite, microdiorite, and lamprophyre dykes and dyke swarms are hosted in the region (Alldrick, 1993).

During the Late Triassic to Early Jurassic, intense ductile deformation occurred in Stuhini Group rocks. This was followed by the Late Jurassic to Late Cretaceous development of Skeena Fold and Thrust Belt. During this period, east-west crustal shortening from collision of the Stikine terrane with the western margin of North America produced north-northwest trending folds and development of a penetrative cleavage, affecting Stuhini Group to Bowser Lake Group rocks. Rocks in the area were subjected to lower greenschist facies regional metamorphism during this time (Febbo et. al, 2019; Alldrick, 1993). Sinistral shearing was active in the Coast Plutonic Complex between 110 Ma – 87 Ma (Febbo et. al, 2019).

Faults are abundant at both local and regional scales in the Stewart area. Alldrick (1993) described five major groups: (1) regional-scale north-striking, subvertical, ductile to brittle faults, (2) northerly-striking moderately west-dipping normal and reverse faults, (3) southeast to northeast striking brittle, subvertical "cross" faults with strong but narrow foliation envelopes and up to a kilometre of lateral offset, (4) decollement surfaces or bedding plane slips near the base of the upper Hazelton Group, and (5) mylonite bands at various orientations and up to a few metres wide at most.

A map illustrating the regional geology and past-producing mines is illustrated in Figure 7-2.

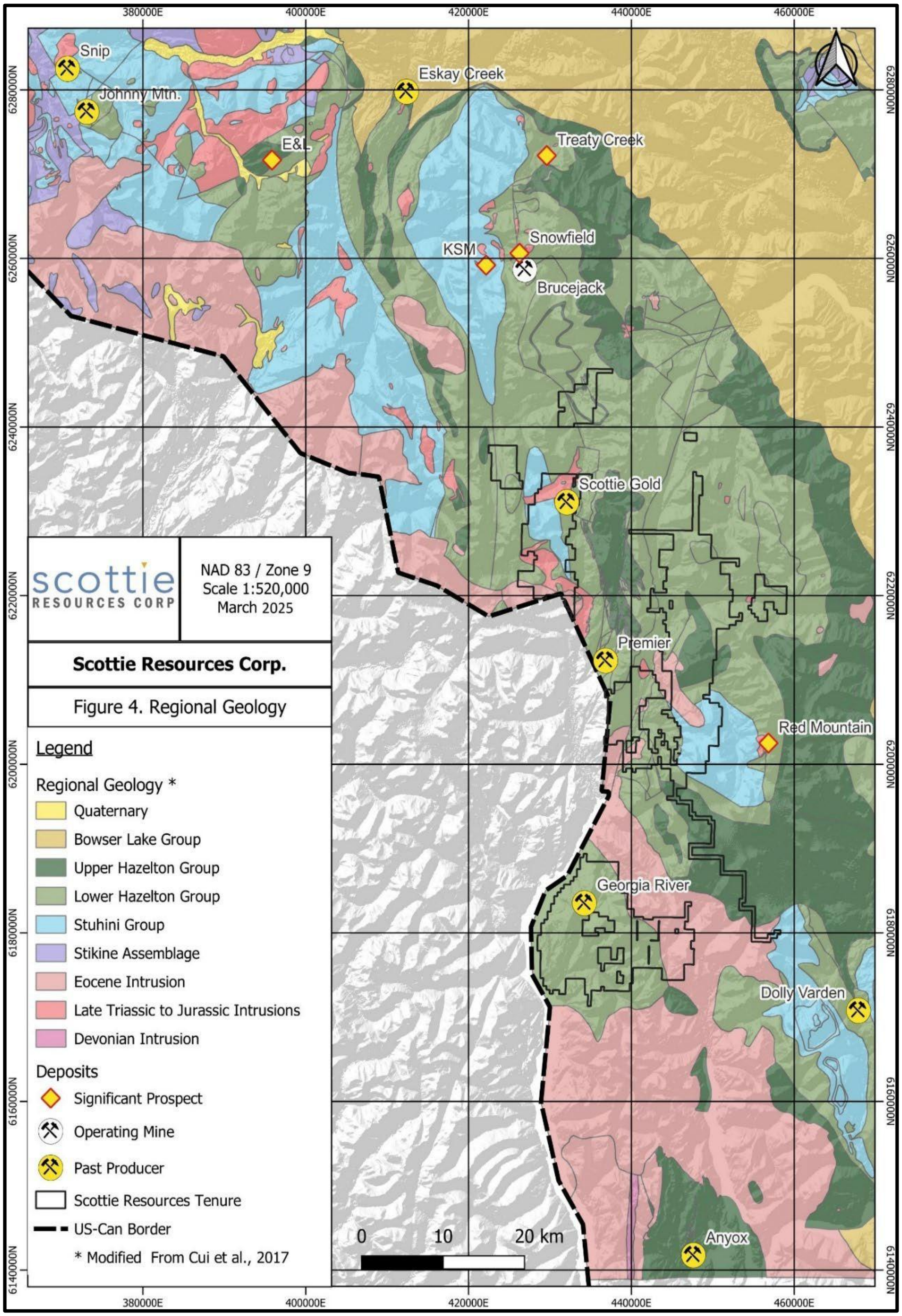


Figure 7-2: Regional Geology of the Scottie Gold Mine Project Area

(Source: Scottie Resources Corp., 2025)

7.2 Property Geology

7.2.1 Lithology

The Scottie Gold Mine Property lies above the volcano-sedimentary rocks of the upper Stuhini and lower Hazelton Groups (Figure 7-3). The rocks in this area have undergone several generations of separate intrusive events, with dykes and stocks of variable ages and compositions seen throughout. During the Cretaceous, east-northeast compression caused the development of north-northwest trending upright folds, resulting in the formation of the Summit Mountain anticline with its Upper Triassic core exposed on the western side of the Property. All stratigraphic units discussed below are taken from Stanley et al. (2022).

The oldest rocks found within the Property are part of the Stuhini group and consist of the lower and upper divisions that lie below the central area of the property, with surface exposures from the Domino area extending down the eastern side of August Mountain and continuing southwards towards the US-Canada border. The lower unit of the lower division is made up of interlayered argillite, limestone, plagioclase-rich fine-grained sandstone, and felsic tuff which grades up section into an upper volcanic unit of augite-phyric trachyandesite breccias, flows and flow-breccias, felsic (trachyte) flows, lapilli- and crystal- lithic tuffs from the upper division. Geochemically the volcanic rocks typically are highly potassic and can have a shoshonitic affinity.

The lowest stratigraphic unit is within the Hazelton group and can be seen at surface west of the Nunatak showing, as an isolated exposure of beige siltstone with lesser intercalated beds of mudstone and sandstone. This isolated exposure lies apparently above lower Stuhini Group sedimentary rocks, across an ice-covered contact.

On the western limb of the Summit Mountain anticline, within the western most portion of the Property, and on the eastern half of the Property, the steeply dipping Betty Creek Formation (lower Hazelton Group) para-conformably lies above the Stuhini volcanic unit. The Betty Creek Formation is comprised of two intervals of feldspar-hornblende-phyric andesitic flows, tuffs, breccias, and conglomerates of the Unuk River andesite unit. Found between these two units there is a thinly bedded sedimentary sequence, comprised of predominantly sandstone, mudstone, and volcanoclastic intervals. This sedimentary sequence, discussed by Alldrick (1993), referred to as the upper siltstone, can be seen at surface in the Stockwork Zone, Blueberry, and on the eastern flank of Summit Mountain (towards August Glacier). The unit is typically subvertical, youngs to the east, and has a north-northwest strike orientation.

The early Jurassic Texas Creek plutonic suite can be found on the northern portion of the Property, it consists of the “Mill porphyry” dykes, Summit Lake stock and isolated plugs in the Blueberry, Scottie Gold Mine and Stockwork areas. The Summit Lake stock can be seen at surface as an elongate, 7 km long, east-northeast trending outcrop, that consists of medium to coarse grained equigranular to porphyritic hornblende and potassium-feldspar rich granodiorite to tonalite. The radiometric dating of the suite returns an age of 185.8 ± 2 to 192.8 ± 2 Ma (Breitsprecher and Mortensen, 2004). In the southern-most area of the Property, intrusives found above the Salmon Glacier within the SW Gossan and Hollywood areas, are shown to be similar in composition to the Summit Stock, with localized megacrystic phases. Further south, towards the U.S-Canada border, an intrusive outcrop of an equigranular biotite ± garnet rich material is seen on the eastern side of Mount Lindaborg and is interpreted to be Eocene in age, belonging to the Coast Plutonic Complex.

There are at least five separate types of dykes seen within the property boundaries. The oldest and earliest intruded are potassium-feldspar phyric to porphyritic dykes, seen with pale grey to white weathering are most likely monzonitic or granodioritic in composition, which may be contemporaneous with or originate from the early Jurassic Summit Lake Stock. These dykes have been mapped in the C Zone, Blueberry and Stockwork areas. Lamprophyre dykes (Jurassic in age) trending west-northwest have been mapped within the Blueberry area. The lamprophyres are typically 2 m wide, tan coloured, porphyritic, and sometimes lightly altered with pyrite mineralization. Alldrick (1993) dated a lamprophyre dyke in the Blueberry Vein area at 186 Ma ± 12 Ma.

A series of post-mineralization tertiary dykes have been mapped throughout the Property. In the Domino and Scottie Gold Mine areas, the youngest are steeply dipping, east - west trending, xenolithic dykes, which are observed to cut the gold bearing shear veins that make up the Scottie Gold Mine deposit. The dykes range from 10 cm to 2 m in width and are host to Summit Stock and volcanic xenoliths within a grey groundmass. Post-dating the xenolithic dykes, are steeply southwest dipping and southeast trending aphanitic to fine-grained pale green microdiorite dykes that can be up to 10 m in width. These dykes are similar in style and orientation to the dykes that cut late Cretaceous to early Tertiary plutons in the region (Rhys, 2006). The youngest dykes within the Property, seen to crosscut both the xenolithic and microdiorite dykes, are a set of dark grey-brown to black mafic dykes of possible lamprophyric composition. These dykes consist of spessartite, fine hornblende phenocrysts and amygdules infilled with calcite. They are similar in appearance to dykes of Oligocene age seen in the region (Rhys, 2006).

Shear-hosted and extensional quartz-carbonate-sulphide rich veins have been recorded in all units on the Property that predate the Cretaceous period and are discussed further in 6.3.

7.2.2 Alteration

The property exhibits widespread regional greenschist metamorphism. The andesitic rocks across the Property typically show pervasive chlorite, minor epidote and trace disseminated pyrite alteration.

Carbonate, silica and sericite are common background alteration minerals seen in the sedimentary and felsic lithologies on the Property.

Mineralized shear-hosted veins are typically associated with an envelope of patchy to homogenous pale green to tan coloured chlorite-sericite-calcite alteration. Pyrrhotite and pyrite may be found as disseminations or as stringers within the alteration envelope. Alteration can extend up to 30 m laterally via principal vein corridors and affect wall rock edges. The narrow shear veins have much more restricted alteration envelopes. Pink calcite can be present locally, when the alteration envelopes are proximal to calcite-rich veins. Localized harder grey sections of alteration within vein envelopes potentially contain potassium feldspar. The boundaries between alteration and shear veins can be gradational without any sharp margins (Rhys, 2006).

Stratigraphic units in contact with the larger intrusions on the Property, such as the Summit Lake stock or the stocks found in the SW Gossan and Hollywood areas, have alteration halos that are <100 m and consist of carbonate-sericite-silica with minor pyrite, with certain areas having localized fine to coarse- grained accessory hornblendes up to 3 cm in length. The Stockwork Zone is host to an eight-hectare surface outcrop of Texas Creek stock intruding andesite and bedded sedimentary rocks. The alteration associated with this contact produced a 500 by 750 m pyrite-illite-muscovite-quartz alteration zone with alteration intensity increasing towards the contact.

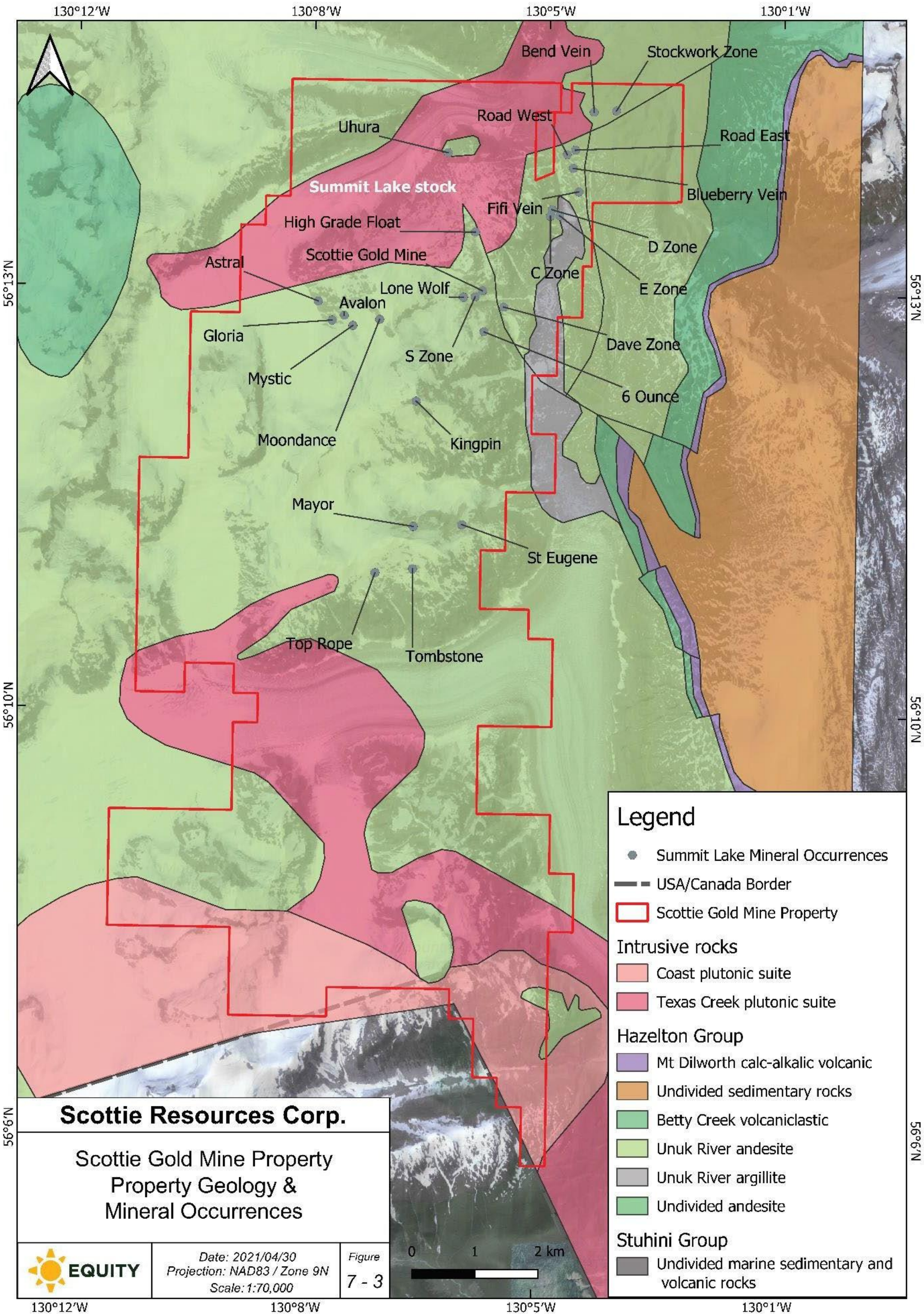


Figure 7-3: Geological Map of the Scottie Gold Mine Project

(Source: Scottie Resources Corp., 2025)

7.2.3 Mineralization

Gold mineralization on the Property is thought to be of intrusion-related gold deposit style. Anomalous gold values occur in shoots that are hosted in veins and replacement zones, with the highest grades typically correlated with increased sulphide content. Base metal and silver values are variable with a few areas on the Property with polymetallic veining producing strongly anomalous silver, copper, lead, and zinc values. However, in gold rich areas such as Blueberry Contact Zone and Scottie Gold Mine area, base metal and silver values are only slightly to moderately elevated but form a much broader footprint than gold mineralization.

Veins on the Property can be characterized as en échelon or sheeted sets of sulphide- carbonate-quartz shear veins and related, but relatively less strained, sulphide- carbonate-quartz extensional veins. These veins and associated mineralization are thought to be derived from Texas Creek intrusions. The Texas Creek plutonic suite is linked to several deposits in the Stewart area, including Premier-Dilworth, KSM and Brucejack.

More than 30 discrete gold +/- silver bearing vein sets have been discovered in the project area with two dominant styles of veining: shear-hosted gold-rich veins with varying sulphide mineralogy and extensional polymetallic silver-rich veins accompanied by a distinct orange iron-carbonate halo. The most advanced and prospective targets on the Property are in the Scottie Gold Mine, Domino, Blueberry and Bend areas, hosting gold-rich shear veins and replacement zones. Mineralization in these areas is discussed individually below.

In addition, fine-grained and likely diagenetic pyrite mineralization, with localized concentrations up to 10%, is commonly found in large gossans within Stuhini argillite units. Sampling of this mineralized sedimentary unit has produced insignificant precious and base metals values.

7.2.4 Scottie Gold Mine Project Area

The Scottie Gold Mine deposit is made up of the Summit Lake vein system, which includes the L, M, N, O, P and the newly discovered Wolf Zone, all of which are hosted within the Unuk River andesite unit. The veins are hosted within shear structures and have been tracked to over 300 m in strike length and in some cases, e.g., the M Zone, up to 400 m of vertical extent. These veins have developed over a collective width of 400 m, with individual structures ranging in width from 2 m to localized areas of 7 m found in the M Zone. Gold-enriched shoots within shear veins typically plunge steeply to the northwest. Relatively unstrained, straight walled, extensional veins generally trend in an east- west fashion and are thought to be splays from the northeast-southwest shear structures.

Both the shear and extensional veins show variable levels of sulphide mineralization, from isolated fine disseminations to large lenses of massive sulphide comprised of mostly pyrrhotite and pyrite with lesser amounts of arsenopyrite, chalcopyrite, galena, gold, sphalerite and tetrahedrite. Veins are generally bordered by chlorite and silica-rich alteration zones that decrease away from the vein into unsilicified-andesite with gangue minerals that include carbonate, sericite, and minor epidote.

Other targets that show similar mineralization and host rocks near the Scottie Gold Mine include the C, D, E, , Dave, and 6 oz. zones, as well as hura, Lone Wolf, and Scottie's Rib.

7.2.5 Domino

The Domino target consists of shear veins that are very similar in composition and morphology to veins seen in the Summit Lake system with the main differences being the widths and the host lithology. Domino is host to veins that are <1 m in width and are hosted within intermediate to felsic volcaniclastics. There are three discrete shear structures with a steep dip and a north-eastern strike that were drill tested in 2020. The mineralization footprint, characterized by alteration, deformation and sulphide mineralization have been tracked over a strike length of 800 m, a vertical extent of over 300 m and a width of 500 m. Elevated gold and silver values are associated with pyrrhotite, pyrite, chalcopyrite, molybdenite, sphalerite, and galena mineralization along with silica, sericite, and chlorite alteration. Proximal targets with similar mineralization to Domino include Moondance, Gloria and Mystic.

7.2.6 Blueberry Contact

The Blueberry Contact zone target is comprised of the near vertical north-south oriented andesite-siltstone contact and numerous moderately northwest dipping veins. The Blueberry Contact Zone is offset by an east-west, dextral fault and can be characterized by north and south portions. Distinct zones comprising wide or high-density Au-Ag-bearing, sulphide-carbonate-quartz veins with associated sericite-chlorite alteration have been defined. The mineralogy of the contact and vein zones can be variable, and can include pyrite, pyrrhotite, sphalerite, molybdenite, galena, and arsenopyrite within quartz-carbonate-chlorite gangue. The typical morphology of the veins are laminated shear veins and/or extensional veins with straight walls and coarse mineral aggregates. The contact zone is comprised by the Road Vein, Blueberry Vein, Lemoffe Vein, and Fifi Vein zones in the north of the deposit and the Gulley and E Zones to the south.

The most consistent gold mineralization along in the Blueberry Contact zone is at the Road zone which has been interpreted as complex, gold-bearing mineralized faults. In some cases, the mineralized faults mark the andesite-siltstone contact. Visible gold is commonly observed in drill core from the Road zone. Road zone veins are characterized by pyrrhotite-pyrite-quartz-carbonate +/- minor chalcopyrite-arsenopyrite-galena-sphalerite with moderate to strong sericite-chlorite alteration.

The Blueberry Vein zone comprises four main parallel mineralized structures within a structural corridor has been interpreted through field and drill core observations. This zone is approximately 60 m wide at surface and an extends to a width of up to 130 m at depth. The four main Blueberry vein structures have been classified as Blueberry Veins 1-4. Blueberry Vein 2 is the primary vein in the Blueberry Vein zone and is a massive-sulphide vein characterized by massive pyrrhotite-pyrite+-quartz- carbonate with rare base metal sulphides and associated sericite-chlorite alteration. The Blueberry Vein 2 is observed to reach 2 m in true thickness.

Approximately 80 m to the south of the Blueberry Vein zone, the Fifi Vein zone, is a structural corridor hosting multiple northwest-dipping veins over a width of 70 m. The three main structures have been identified as Fifi Veins 1-3 and are characterized by massive pyrrhotite-pyrite+-quartz+- carbonate with rare base metal sulphides and associated sericite-chlorite alteration.

Approximately 25 m southwest of the Fifi Vein zone, another structural corridor has been identified in the vicinity of the A Portal known as the Lemoffe Vein zone. At least three mineralized shears have been intersected over a width of 45 m and are characterized by massive pyrrhotite-pyrite+- quartz+-carbonate with rare base metal sulphides and associated sericite-chlorite alteration.

The Gulley zone is located west of the C Zone and is characterized by intervals Au-bearing quartz-carbonate-pyrrhotite-pyrite veins and strong sericite-chlorite alteration along the andesite-siltstone contact.

The E Zone is located 230 m south of the Gulley zone and is characterized by Au-Ag-sulfide mineralization, strong sericite-chlorite alteration along the andesite-siltstone contact and a massive sulphide northwest dipping structure (The Serac Vein). The Serac Vein is characterized by a ~2 m wide (true width unknown) pyrrhotite vein with minor pyrite-arsenopyrite-chalcopyrite-galena-sphalerite and quartz-carbonate gangue with associated sericite-chlorite alteration. Immediately underlying the Serac Vein are smaller base-metal sulphide extensional veins which are hosted in the siltstone unit. The base-metal signature in the E Zone is elevated compared to other zones on the Property.

7.2.7 Bend

The Bend Vein is a quartz-carbonate-chlorite-sulphide shear vein system with an average true width of 1.7 m. The vein can be found within or proximal to the east-northeast trending Bend Fault, a 700 m long structure within the Unuk River andesite. The Bend Vein strikes at 060˚ and dips northwest at around 45-70˚, with the high-grade gold and silver concentrations found in the west plunging shoot. The sulphide mineralization within the vein includes pyrite, pyrrhotite, chalcopyrite, sphalerite, galena, molybdenite, and cobaltite. Crude lamination of sulphides and gangue minerals are theorized to have been produced via multiple stages of shearing and mineralization within the Bend Fault. The footwall of the Bend Vein has been brecciated by late-stage faulting. The vein swells and pinches both along strike and dip. The vein has been tracked 110 m along strike and 80 m vertically.

7.2.8 Stockwork

The Stockwork zone is located several hundred metres east of the Bend Vein and is comprised of a 750 by 500 m zone of quartz-sericite-pyrite alteration centred by a quartz vein stockwork. The zone contains pyrrhotite, pyrite, trace molybdenite and trace chalcopyrite, along with anomalous gold values. Broad elevated Au intervals and anomalous Mo in the Stockwork zone suggests the presence of porphyry-style mineralization.

7.2.9 C and D Zones

The C and D Zones have a style of mineralization that is similar to the Scottie Gold Mine deposit and are also hosted in the Unuk River andesite. C Zone veins are hosted in shear structures that have been delineated over 270 m in strike length and up to 150 m in vertical extent. Due to the paucity of drilling, the extent of the D Zone vein is much more limited with a strike length of ~130 m and vertical extent of 90 m. Shear and extensional veins show a range in sulphide content from isolated disseminated grains to lenses of massive sulphide comprising mostly pyrrhotite and pyrite with lesser sphalerite, chalcopyrite, galena, arsenopyrite, and gold. Veins contain quartz-carbonate gangue and are bordered by siliceous and chlorite-sericite-rich alteration zones.

8.0 DEPOSIT TYPES

Mineralization at the Scottie Gold Mine Property consists of sulphide-rich shear veins and extensional veins that are part of the intrusion-related gold deposit type. These deposits are transitional between deeper porphyry and shallower epithermal deposits and are sometimes referred to as mesothermal veins. Other examples of this deposit type in British Columbia include the past-producing Snip and Le Roi deposits. The below description of this deposit type is based on Alldrick (1996).

Pyrrhotite-rich intrusion-related veins consist of parallel tabular to cymoid arrays emplaced around the periphery of a causative subvolcanic intrusion. Individual veins range from centimetres to metres in width and can be traced for up to hundreds of metres along strike. Mineralization is controlled by faults and shear zones that are spatially associated with porphyritic intrusions and, in some cases, mineralized porphyries.

The intrusion-related veins typically develop in oceanic and continental margin settings. Host rocks consist of intermediate volcanic rocks, marine sedimentary rocks, and/or earlier intrusive phases to the causative intrusion. Veins consist mostly of quartz, carbonate, pyrrhotite and pyrite, with localized pods of massive to semi-massive sulphide passing outwards into quartz- and/or carbonate- dominant shear veins. Mineralization mineralogy consists mostly of pyrrhotite and pyrite with minor native gold, electrum, and base metal sulphides (e.g. chalcopyrite, galena, sphalerite). Besides quartz and carbonate, gangue mineralogy also includes chlorite, sericite, K-feldspar, and/or biotite.

Wallrock alteration extends from several centimetres to metres into the host rocks, consisting mostly of chlorite, sericite, pyrite, carbonate, biotite, epidote, and/or K-feldspar.

Mineralization is interpreted as syn-intrusive and formed within the thermally-controlled brittle-ductile envelope that surrounds the causative intrusion.

9.0 EXPLORATION

The following exploration activities have been conducted on the property by Scottie Resources. This is a summary of material presented in the Assessment Reports submitted by Scottie Resources Corp. from 2019 through 2024.

9.1 2019

Property-wide geological mapping was done along with the collection of 444 rock, 14 soil, 27 tailing, and 5 stream sediment samples. Mapping and prospecting identified numerous new showings with rock sampling returning up to 536 g/t Au and 5,380 g/t Ag (Guestrin, 2019; Guestrin 2021).

9.2 2020

2D induced polarization geophysical surveys were done at the Domino, Scottie's Rib, and Bend targets, and an airborne mag and EM survey completed between the Salmon Glacier and the northern claim boundary. In addition, infill sampling was completed over many prospects as a follow-up to the 2019 sampling, with a total of 905 rock and 14 sediment samples collected.

9.3 2021

1.08 km^2 of 3D DC-resistivity and induced polarisation (DCIP) were completed at the Blueberry and Domino zones, along with 1,560 m of borehole TEM surveying carried out at the Scottie Gold Mine.

9.4 2023

A prospecting and sampling program was done across the Scottie Gold Mine Project. The program targeted multiple zones, including the C and D Zones, Blueberry, Gulley, Serac, E Zone, and the High-Grade Float Zone. The highest-grade sample collected during the program returned 21.6 g/t gold from a grab sample located approximately 1 km northwest of the SGM.

9.5 2024

A prospecting and sampling program was done focused on zones located northwest and southeast of the Scottie Gold Mine: The E Zone, Golden Buckle, High Grade Float and Scottie East Zone. A total of 48 rock samples were taken. The following figure details the 2024 prospecting program.

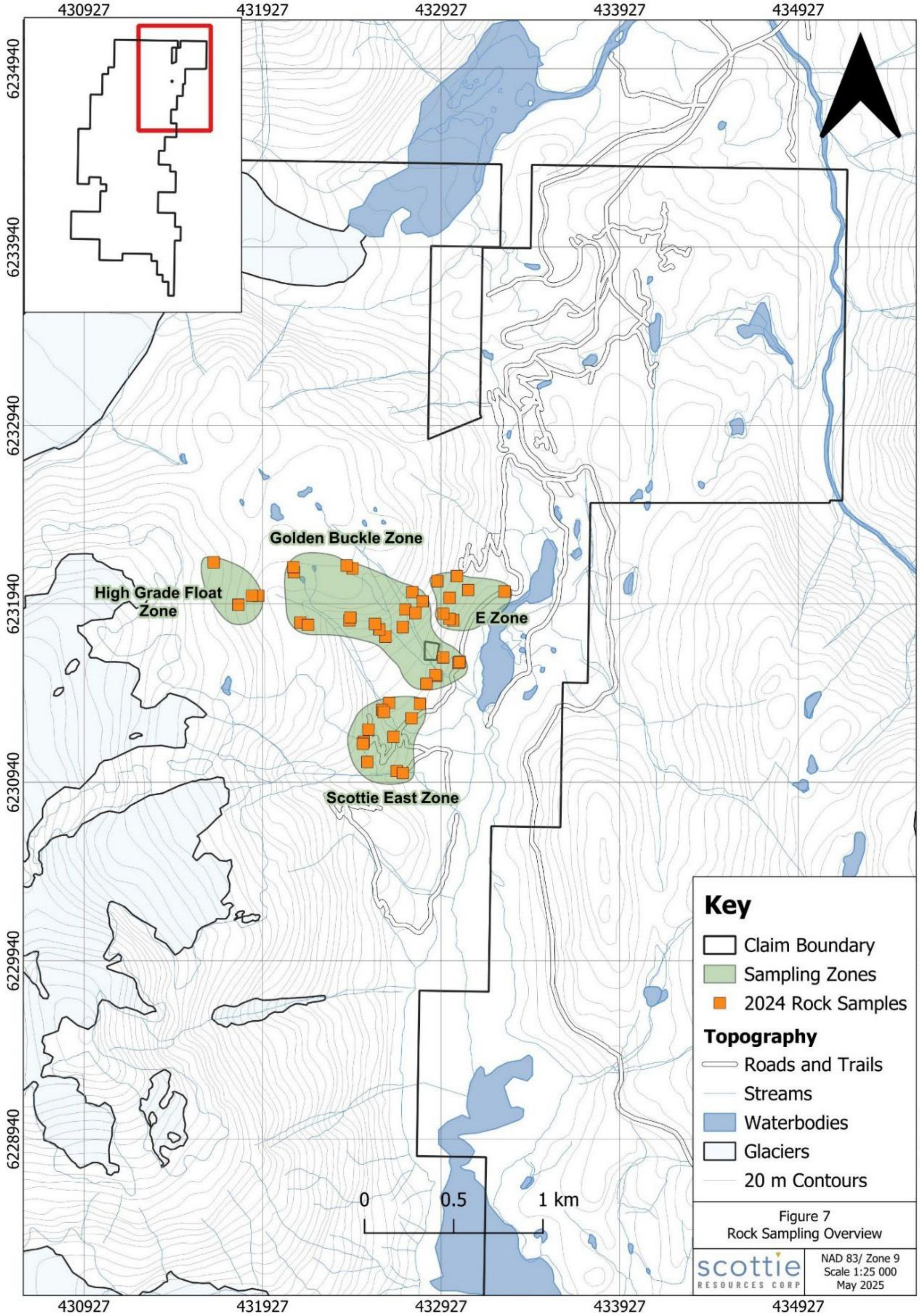


Figure 9-1: 2024 Rock Sampling Overview with Sampling Zones

(Source: Scottie Resources, 2024)

10.0 DRILLING

Drilling by Scottie Resources Corp. from 2016 through 2024 is summarized in this section. Table 10-1 is a summary of the drilling done by Scottie Resources with Figure 10-1 showing a plan map of the drilling at the Blueberry area deposits (which include Bend and Gulley) and Figure 10-2 illustrating the drilling to date at the Scottie Mine deposit. The majority of drilling at the Blueberry, Bend and Gulley deposits has been done since 2019 whereas at the Scottie deposit there is significant historic drilling. Figure 10-3 is a three-dimensional view of the Scottie Mine drilling since 2019 and the mineralized zone shapes modelled for the resource estimate. This illustrates the good coverage of recent drilling that has been used to validate the historical drilling, as discussed in Section 12.

Table 10-1: Summary of Diamond Drilling within the Database and within the Resource Domains

Area	Year	# DDH	Length (m)	Within the Database		Within the Domains	
				# Assays	Total Assay Length (m)	# Assays	Total Assay Length (m)
Total Database	Undefined	222	24,516.7	0	0.0	0	0.0
	1948	19	1,627.6	62	59.0	29	26.4
	1979	22	932.1	226	135.2	71	39.3
	1981	28	588.2	123	139.3	24	24.8
	1982	67	4,139.2	719	615.6	173	141.1
	1983	68	6,886.0	1,372	877.8	273	162.0
	1984	117	8,169.0	1,440	938.6	442	291.9
	1987	20	1,975.3	263	194.9	47	33.6
	1990	1	84.4	14	19.7	0	0.0
	1991	10	306.4	93	50.6	0	0.0
	2004	14	1,273.8	501	505.7	40	31.5
	2005	45	3,809.3	1,226	1,227.0	215	168.5
	2006	16	2,573.5	536	572.9	81	66.6
	2016	19	2,158.6	691	857.6	0	0.0
	2019	19	2,033.5	866	1,652.2	108	127.5
	2020	46	7,054.7	3,748	4,934.6	386	339.5
	2021	78	12,857.8	8,412	11,383.6	1,220	1,095.5
	2022	89	17,159.5	11,913	16,503.6	2,017	1,986.4
	2023	84	20,167.6	9,777	19,521.8	1,670	1,742.5
	2024	44	10,270.0	5,397	10,108.2	597	677.0
	Total	**1,028**	**128,583.2**	**47,379**	**70,297.7**	**7,393**	**6,954.1**

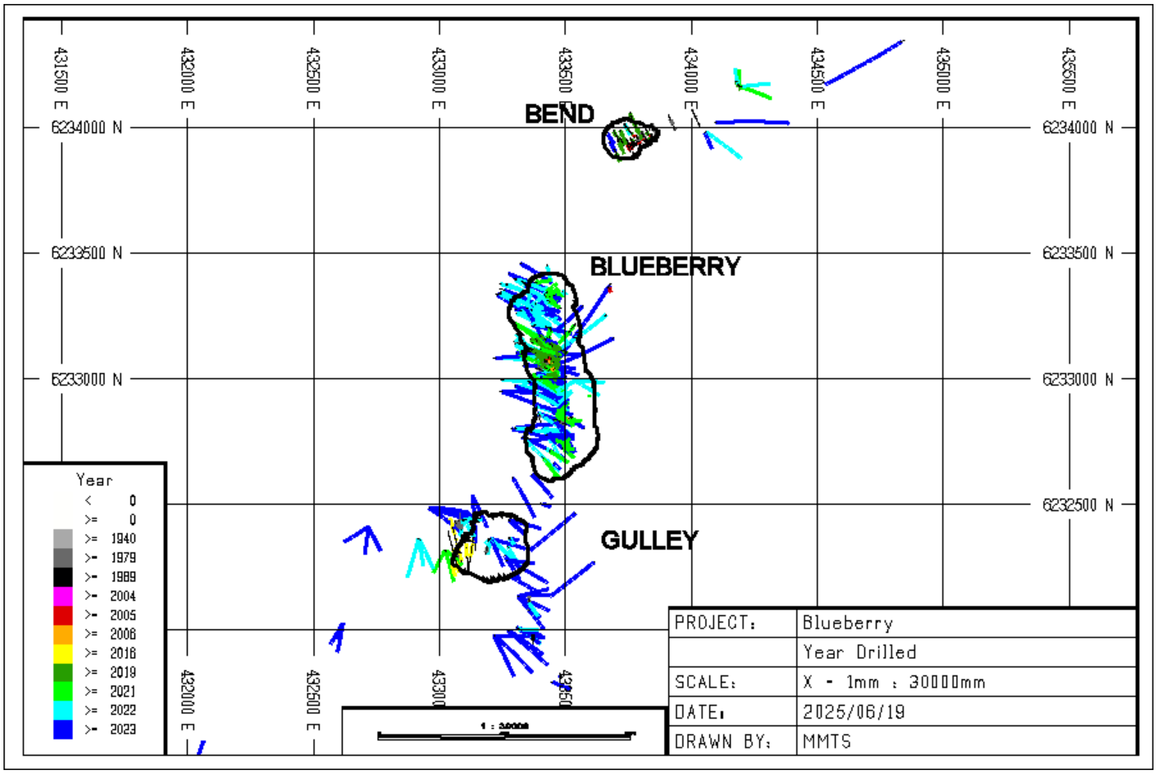


Figure 10-1: Plan View of Drillholes by Year – Blueberry / Bend / Gulley

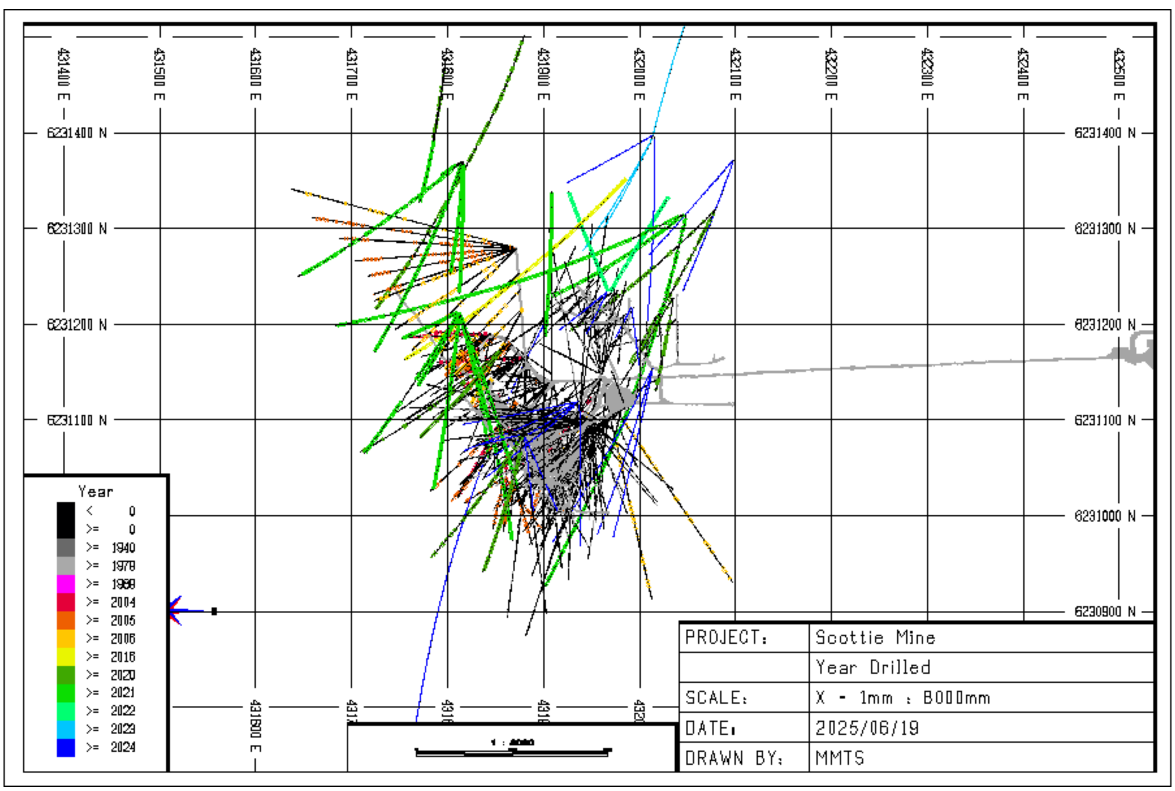


Figure 10-2: Plan View of Drillholes by Year – Scottie Deposit

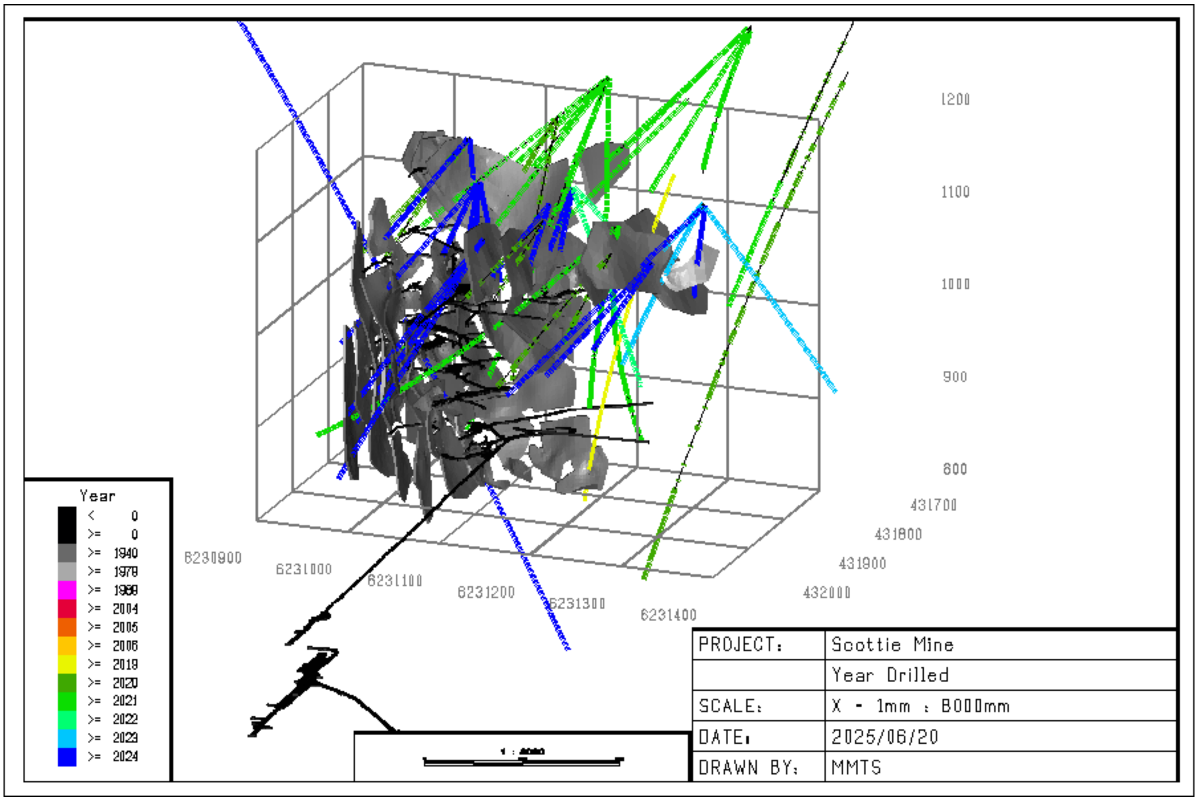


Figure 10-3: 3D View of Drilling from 2019- 2024 at the Scottie Mine Deposit

The following sections provide summaries of the drilling done by Scottie Resources Corp. from 2019 through 2024, extracted from the assessment reports for these years. Figure 10-4 is a map showing the main areas drilled by Scottie Resources.

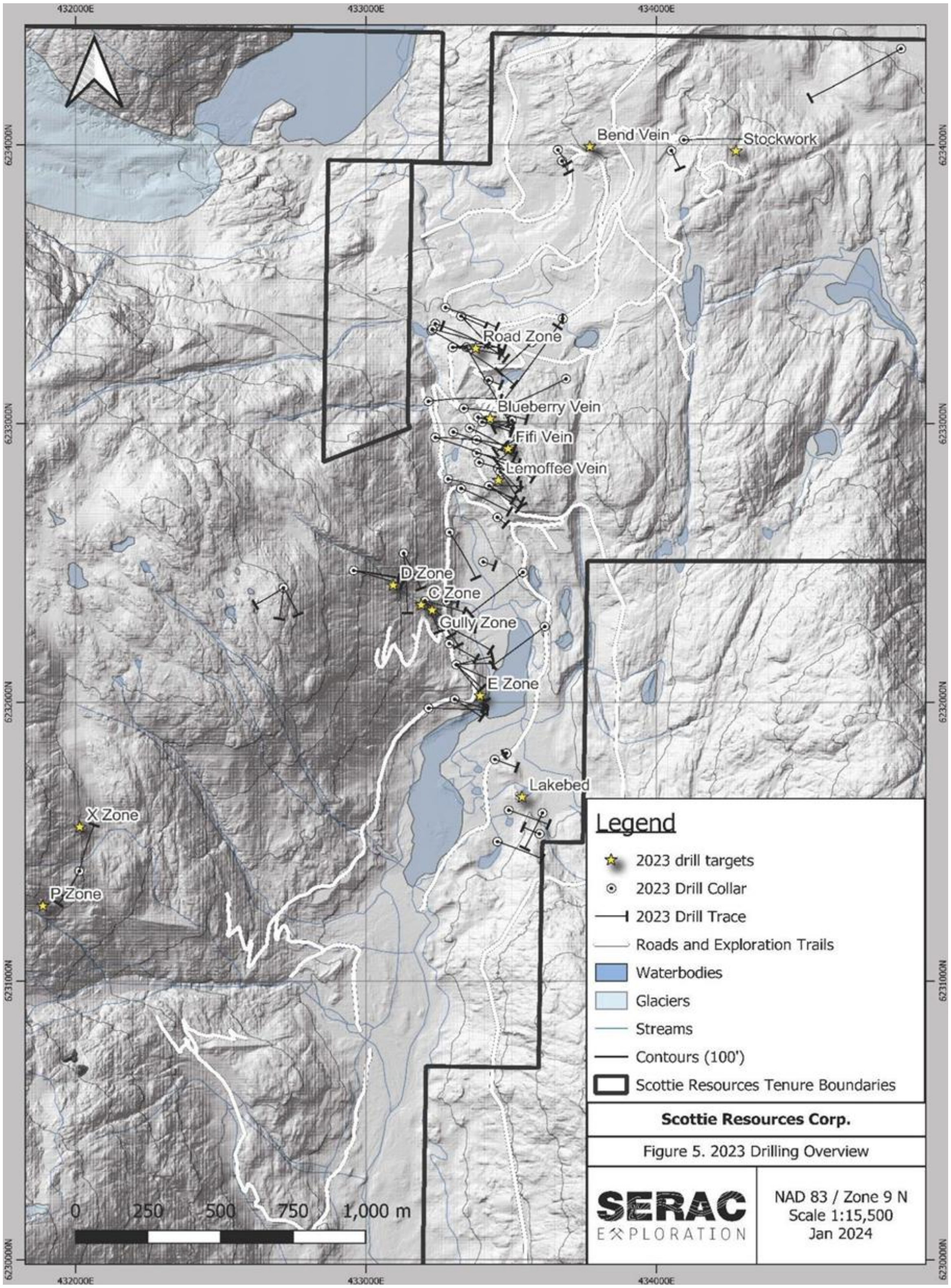


Figure 10-4: Map of Main Drilling Areas

(Source: Serac, 2023)

10.1 2016 through 2020 Drilling

The following is an excerpt edited from the 2021 NI43-101 Technical report.

Scottie Resources operated three diamond drilling campaigns on the Scottie Gold Mine Property, first as Rotation (2016) and then as Scottie Resources in 2019 and 2020. The bulk of drilling has been done with skid- and helicopter-portable diamond drilling rigs, with methods described here.

The 2016 drill program was carried out from 1 June and 31 October 2016 by Sunbeam Drilling of Stewart, BC, using a B-10 underground drill and a JKS drill with a B-10 drill head (Kruchkowski, 2017). The drill program was managed by Rotation and comprised 21 holes for 2,648 m of BTW sized core, with 18 holes for 1,935 m drilled on the C and D zones and the remaining 3 holes (713 m) testing other targets. No downhole surveys were completed, no geotechnical parameters were measured, and no post-drilling differential GPS (DGPS) surveys were done so that final hole positions may have location errors of up to 10 m. Logged features include only vein, alteration, and mineralization occurrences.

The 2019 drill program was completed from September 15 to October 13, 2019, by Driftwood Drilling Ltd ("Driftwood") of Smithers, BC, using one SRS 3000 skid-mounted and one SRS 3000 helicopter-portable diamond drill. The drill program was managed by Equity Exploration and comprised 20 holes for 2,050 m of NQ core, with most holes ranging between 35 m to 162 m in depth except for one hole drilled to 539 m on the M Zone. Holes were spotted with a handheld GPS and aligned with a compass. Downhole surveys were done with a Reflex EZ-Shot. Average recovery (94%) is high by industry standards whereas RQD is on average fair to good (74%). A post-drilling differential GPS (DGPS) survey was completed in 2020 to improve the accuracy of collar locations to <0.1 m. Core was placed into wooden trays at the drilling site, then transported to the core processing facility by either pickup truck or helicopter. Logged features include lithology, alteration, mineralization, structures, and veins. No specific gravity data was collected.

The 2020 drill program was also managed by Equity Exploration and completed by Driftwood, using one SRS 3000 skid-mounted and two SRS 3000 helicopter-portable diamond drills. A total of 46 holes were completed for 7,055 m of NQ core, with holes ranging from 43 to 713 m in depth. Holes were spotted with a handheld GPS and aligned with a DeviAligner north-seeking gyro system. Downhole surveys were done with DeviShot and DeviGyro tools. Average recovery (97%) is high by industry standards whereas RQD is on average good (83%). A post-drilling differential GPS (DGPS) surveys was done to improve the accuracy of collar locations to <0.1 m. Core was placed into wooden trays at the drilling site, then transported to the core processing facility by either pickup truck or helicopter. Logged features include lithology, alteration, mineralization, structures, and veins. No specific gravity data was collected.

10.1.1 Scottie Gold Mine

Scottie drilled 13 holes into the Scottie Gold Mine area for a total of 3,821 m. Hole depths averaged 294 m within a range of 144 m to 713 m and were mostly drilled between azimuths of 190° to 230° and dips of -45° to -65°. The two holes drilled in 2016 were collared 400 m to 500 m east-southeast of the Scottie Gold Mine workings and were drilled at orientations of 310°-340° and dips of -45°. A total of 1,321 core samples were taken for 1,777 m, covering 47% of metres drilled by Scottie Resources at the Scottie Gold Mine.

10.1.2 Blueberry

Nineteen holes have been drilled into the Blueberry area during this time by Scottie for a total of 2,258 m, with eight of these drilled in 2019 (633 m) followed by 11 holes in 2020 (1,625 m). All these holes were drilled in and around the historical Blueberry Vein showing although most of them targeted replacement-style gold mineralization on the Contact Zone. Holes were drilled to an average depth of 119 m, within a range of 42 m to 258 m, between azimuths of 100° to 150°, and dips of -45° to -65°. A total of 1,301 core samples were taken from the Blueberry holes, covering 1,828 m of the 2,258 m drilled (81%).

10.2 2020 Drilling

Forty-six diamond drillholes, totaling 7,055 m, were completed between July and September 2020. The drilling was focused on the Bend Vein, Blueberry Vein, Scottie Gold Mine, 6 oz. Zone, and Domino targets.

A total of 11 drillholes were drilled at Blueberry targeting the Blueberry Vein and mineralization along the AND-SLT contact. 10 holes were drilled in the Scottie Gold Mine, targeting the M, N, O, and P zones. Four holes were drilled at Bend Vein. 18 holes were drilled at Domino, and three holes were drilled at 6 oz. Zone.

10.2.1 Blueberry Vein

Eleven drillholes for total of 1,624.7 m were drilled at the Blueberry Vein target. Three holes (SR20-21 to SR20-23) were drilled to test down dip and along strike of the Blueberry Vein. Two holes (SR20-40 and SR20-45) were drilled ~50 m south of the Blueberry Vein beneath a high-grade intercept (>15 g/t Au) near the andesite-siltstone contact. Six holes tested for mineralization within a north-plunging structure along the andesite-siltstone contact.

10.2.2 Scottie Mine Area

Ten drillholes were for a total of 2,815 m were drilled at the Scottie Gold Mine. Six holes tested the M Zone. Four holes tested the O Zone. Several drillholes were designed to test the M Zone were dual purpose by also testing the N and L Zone veins. Similarly, drillholes targeting the O Zone also included tests of the P Zone and interpreted east-west oriented veins related to the O Zone.

10.2.3 Domino

18 holes for a total of 1,979 m were drilled at Domino. drilling targeted northeast-southwest oriented sulphidized shear veins that returned high-grade gold results from 2019 and 2020 prospecting campaigns. Four drillholes tested Domino North, six drillholes tested Domino East, five drillholes tested Domino Central, and three drillholes tested Domino West.

10.2.4 6 oz. Zone

Three holes for a total of 321 m were drilled at 6 oz. Zone. Drilling targeted beneath sulphidized shear veins that returned high-grade gold results from 2019 and 2020 prospecting campaigns. Mineralization at 6 oz. Zone is characterized by polymetallic sulphidized veins hosted within andesitic host rock, typical in form to other targets throughout the Property.

10.3 2021 Drilling

Drills were aligned using Survey Tech's Devi li gner and a DeviShot downhole survey tool collected dip and azimuth readings every 30 m. Core orientation was completed using the DeviCore BBT system. Upon completion of the hole, a DeviGyro downhole tool was used to survey the entire hole and an RTK receiver was used to collect the easting, northing, and elevation of each collar.

Drill core was transported from each drill site and quick-logged at the Granduc Mill camp before being transported to Scottie Resources' logging facilities in the town of Stewart for logging and sampling. Lithology, alteration, mineralization, veining, oxidation, structure, oriented core, recovery, specific gravity, and magnetic susceptibility data were collected during the geological and geotechnical logging process. Prior to sampling, photographs were taken after the core was marked and tagged.

At the Blueberry, Stockwork, and C Zone areas, holes were sampled from top to bottom. Selective sampling was completed at the Scottie Gold Mine and Domino areas where gold and silver are known to be restricted to altered and sulphidized shear zones. Half-core sampling was conducted with a diamond saw at the core logging facility.

ollow ing logging and sampling, core was banded and transported to Yellowhead Helicopter's Bitter Creek staging area approximately 15 km north of Stewart for storage.

10.3.1 Stockwork Zone

Two holes were drilled for a total of 275.5 m were drilled at the Stockwork Zone. SR21-085 was designed to test the andesite-bedded sedimentary unit contact to assess whether a gold bearing structural trap similar to Blueberry exists in this portion of the Property. SR21-087 was drilled to test a quartz-sericite- pyrite alteration zone accompanied by a 500 m by 300 m gold-in-soil anomaly outlined from 1983, 1990, and 2006 geochemical sampling campaigns. Both holes intersected pervasive weak to moderate silica and sericite alteration accompanied by broad intervals of elevated gold. No significant base metal values were returned.

SR21-085 was collared in andesite and intersected a bedded siltstone unit at 56.0 m. The contact between the two units appeared to be fault-bounded and is interpreted to be the eastern extension of the Bend Fault. Proximal to the contact, anomalous gold of greater than 1 g/t was intersected in both units.

SR21-087 was collared into andesite and intersected a diorite intrusion at 167.64 m. Once in the intrusion, gold values were less than 100 ppb with elevated molybdenum values up to 86 ppm.

10.3.2 Blueberry

The Blueberry target is comprised of the Road North, Blueberry Vein, Grizzly, and Lemoffe zones. A total of 41 holes were drilled across all four zones for a total of 4,915.9 m. Drillholes were generally designed to test both the north-south oriented andesite-siltstone contact, a target interpreted to be a gold-bearing structural trap identified in 2019 to 2020 drilling and rock sampling, and east to northeast trending mineralized cross-structures.

Drilling in 2021 was successful in intersecting numerous gold-rich structures. Gold mineralization has been interpreted to be found in three different types of structures:

1. Andesite-Siltstone Contact

Contact Intercepts along the andesite-siltstone contact have returned narrow to broad, high-grade intervals with gold found in both lithological units directly at, or proximal to the contact. A total of 37 of the 39 holes that intersected the andesite-siltstone contact were altered or mineralized at the contact, with 24 of these holes assaying greater than 1 g/t Au proximal to the contact.

Drilling in 2021 expanded the strike length of gold mineralization along the contact with significant intercepts produced within all zones at the Blueberry target over a total strike length of approximately 720 m. Each of the Road North, Blueberry Vein, Grizzly, and Lemoffe zones host a north-plunging ore shoot that appears to be defined by the intersection lineation between northeast structures and the lithological contact.

SR21-131 identified the potential of the contact zone with an intercept of 34.56 g/t Au and 3.34 g/t Ag over 11.86 m at Road North. In addition to delineating the structure along strike, drilling in 2021 expanded the known depth of contact associated gold to 225 m in the Blueberry Vein area with SR21-138 returning 15.26 g/t Au and 9.11 g/t Ag over 13.49 m. The most consistent gold mineralization along the contact is at the Road zone where there appears to be more complex faulting in the vicinity of the drilled area.

2. Northeast Trending Veins

Numerous holes intercepted northwest-dipping, east-northeast to northeast trending veins that assayed greater than 1 g/t Au. The most significant results for this style of mineralization are found at the Blueberry Vein, Grizzly, and Lemoffe areas.

The Blueberry Vein was the initial showing in the Blueberry area and the focus of drilling between 1983 and 2005 until the potential of the andesite-siltstone contact was first realized. The Blueberry Vein itself is a massive sulphide shear vein exceeding 2.0 m in width with multiple parallel mineralized structures mapped within a structural corridor totaling approximately 15 m of width. Drillholes SR21-069, SR21-070, SR21-071, SR21-089, SR21-090, SR21-136, SR21-138, and SR21-141 were designed to intersect the Blueberry Vein in conjunction with testing the andesite-siltstone contact. Results from the Blueberry Vein structural corridor were variable, with the best results from near surface intercepts in the central part of the structure, which includes 3.21 g/t Au and 2.70 g/t Ag over 10.0 m from SR21-070. Negligible gold values were returned from intersections at its southwestern extent and at depth.

Approximately 225 m to the south of the Blueberry Vein at the Grizzly area is a structural corridor hosting multiple northwest-dipping veins over a width of 70 m. Scottie Resources first identified this zone in 2020 with rock sampling results of up to 37.0 g/t Au. Holes that intercepted mineralized cross-structures at Grizzly are SR21-076, SR21-077, SR21-078, SR21-121, and SR21-123 with the best intercept of 22.1 g/t Au and 12.0 g/t Ag over 0.85 m from SR21-123.

Approximately 100 m southwest of the Grizzly, a structural corridor has been identified in the vicinity of the A Portal known as the Lemoffe veins. Multiple mineralized shears have been intersected over a width of 75 m. This area has been tested by holes SR21-072, SR21-073, SR21-075, SR21-130, and SR21-135. The best intercepts in this area are 125.0 g/t Au and 35.9 g/t Ag over 0.29 m from SR21-072 and 4.27 g/t Au and 4.32 g/t Ag over 11.9 m (not true-width) from SR21-075. In addition to these high-grade intercepts, SR21-130 demonstrates the potential of intersecting zones of higher vein density with a broad lower-grade interval of 1.66 g/t Au and 2.74 g/t Ag over 32.96 m.

Furthermore, several isolated structures of similar orientation were intersected between the Road and Blueberry Vein areas.

3. Contact Parallel Structures

In 2014, Decade Resources identified the potential of north trending veins with the discovery of the siltstone hosted Big M Vein where trench sampling returned up to 3,418.07 g/t Au (Decade Resources, 2014). In 2021, several intercepts within the siltstone not proximal to the contact returned anomalous gold in holes SR21-070, SR21-090, SR21-126, and SR21-129. Previous detailed geological mapping by Scottie Resources has identified multiple contact-parallel faults in the Blueberry area, which could be the source of the intercepts, however this is currently speculative given that oriented core measurements were not collected at these intercepts. Further, the 3D DCIP survey over the Blueberry area shows that anomalous conductivity-high and chargeability-low data does highlight the contact zone, and that several other north-south oriented linear anomalies are present to the east of the contact.

10.3.3 C Zone

Three holes were drilled at the C Zone for a total of 450.9 m. All three holes were designed to test the western extension of the C Zone's main showing where chip sampling in 2020 returned 26.21 g t u over 7.0 m.

Each hole intersected multiple mineralized veins and shear structures hosting pyrrhotite-pyrite ± chalcopyrite-arsenopyrite-sphalerite. Most veins intersected are narrow (sub metre) except for a 1.96 m interval in SR21-118 that returned 5.89 g/t Au and 22.2 g/t Ag. Despite the narrow widths of mineralized structures, there appears to be potential for broader gold-bearing intervals with higher vein density. SR21-116 intersected multiple closely spaced shears returning 1.02 g/t Au and 1.42 g/t Ag over 24.11 m.

10.3.4 Scottie Gold Mine

Drilling at the Scottie Gold Mine target consisted of 17 holes for a total of 5,179.67 m. The main objective of drilling at the Scottie Gold Mine was to expand the mineralized extent of the different zones along strike and at depth rather than focus on in-fill drilling. Holes were designed to test multiple zones and step-out from the defined areas of the P, O, M, N, and L Zones, with step-outs of at least 30 m from previous intercepts.

Results from each zone are discussed individually below.

10.3.4.1 P Zone

Drilling at the P Zone was successful in intercepting near-surface gold-bearing structures with 275 m of strike length tested in 2021. The eastern portion of the P Zone, which is partially obstructed on surface by till, was intersected by holes SR21-094, SR21-096, and SR21-099, each of which returned anomalous gold. Drilling consisted of 15-35 m step outs to the west from significant intercepts in 2020, such as SR20-65 which returned 10.7 g/t Au and 31.2 g/t Ag over 2.77 m. The best intercept of these three holes from 2021 is 11.78 g/t Au and 12.58 g/t Ag over 6.57 m from SR21-094. Approximately 125 m west along strike of SR21-096, the central portion of the P Zone was tested by SR21-124 which intersected 7.53 g/t Au over 0.84 m.

The western portion of the P Zone, 225-250 m along strike to the west of SR21-096, was tested by SR21-093, SR21-102, SR21-105, SR21-108, SR21-109, SR21-113. On the surface, P Zone West hosts a series of pyrrhotite bearing veins over a width of 40 m. Relative to other zones, P Zone West generally has lower vein density with individual veins usually less than 1.0 m in width. Drilling at P Zone West was successful in intersecting multiple veins. However, as consistent with surface samples from previous sampling campaigns, gold values were low with the best intercept of 0.44 g/t Au over 1.99 m from SR21-113.

10.3.4.2 O Zone

The O Zone was intersected by holes SR21-094, SR21-096, and SR21-099 with drilling designed to test its northwest and southeast extensions, as well as down-dip extensions. All three holes intersected anomalous 32 gold, extending the strike length and down-dip extensions of the O Zone, with the best intercept of 20.62 g/t Au and 10.95 g/t Ag over 2.22 m from SR21-096.

10.3.4.3 M Zone

Drillholes at the M Zone were designed to test northwestern and southeastern extensions, as well as infill between shallowly plunging shoots interpreted from 2020 drilling. The best intercepts of the M Zone were across its upper shoot beneath the Skye Zone, which was tested in 2020 with encouraging results from two drillholes, including SR20-52 with 4.69 g/t Au and 2.42 g/t Ag over 15.48 m. SR21-095 produced the strongest results of 2.2 g/t Au and 6.05 g/t Ag over 10.09 m and 37.17 g/t Au and 8.6 g/t Ag over 3.71 m from two vein sets spaced less than 20 m apart. At 35 m downdip of SR21-095, the second-best intercept at the M Zone in 2021 was 4.03 g/t Au and 1.68 g/t Ag over 22.01 m, including 9.03 g/t Au and 3.08 g/t Ag over 7.39m, from SR21-097.

Beneath the Skye Zone hits, SR21-100 tested the down-plunge extension of an interpreted 2nd steeply plunging shoot within M Zone resulting in 6.68 g/t Au ad 10 g/t Ag over 1.0 m.

The southeastern extent of the M Zone was intersected by SR21-099 within a gap of gold hits produced from 1982-1984 drilling. Despite intersecting 10.68 m of veining within altered and mineralized andesite, no significant assays were produced.

Most of the drilling at M Zone was completed at its northwestern extension which was interpreted to be open along strike with good potential for expansion of the deposit. SR21-096 tested the lower northwestern extension in an underexplored area with only limited drilling in 2005, intersecting 3.42 g/t Au and 4.03 g/t Ag over 1.95 m. The interpreted upper northwestern extension of the M Zone has never been drilled prior to 2021 and was tested by Drillholes SR21-102, SR21-103, SR21-106, SR21-109, SR21- 114, and SR21-122. All holes intersected intervals of altered and mineralized andesite with veining that could be interpreted as part of the M Zone; however, results generally returned low gold values. The best intercept from the upper northwestern extension is from SR21-103, which returned 8.89 g/t Au and 16.65 g/t Ag over 0.97 m.

10.3.4.4 N Zone

Drilling at the N Zone was designed to test its northwestern and southeastern extensions and was successfully intersected by SR21-097, SR21-099, and SR21-100, with the best intercept on its southeastern extent by SR21- 099 of 4.28 g/t Au and 5.80 g/t Ag over 14.43 m, including 14.45 g/t Au and 15.06 g/t Ag over 3.65 m. Drilling in this portion of the N Zone is limited and open in multiple directions. The northwestern extent also returned encouraging results, with SR21-100 intersecting 5.56 g/t Au and 10.8 g/t Ag over 2.5 m across a gap in previous drilling and SR21-097 hitting multiple parallel mineralized zones with the best intercept of 2.0 m of 0.83 g/t Au.

SR21-095 was also drilled to test the northwestern upper extent N Zone but intersected 42 m of microdiorite dyke at its projected intercept before the hole was shutdown. Holes SR21-096, SR21-114 and SR21- 122 intersected vein zones that could be interpreted as the northwestern and up-dip extensions of N Zone but returned no significant assay results. The best intercept of these holes is from SR21-096, which returned 0.46 g/t Au and 53.0 g/t Ag over 1.9 m. More drilling is required in this area to confirm whether these mineralized zones are associated with the N Zone.

10.3.4.5 L Zone

The L Zone saw limited drilling in 2021 and was only intersected by two drillholes. Its deep position in the mineralized system at the Scottie Gold Mine requires deep holes to test this structure when collared at surface. SR21-097 tested the up-dip extension of the L Zone between a cluster of hits from underground drilling in 2005. Assays results were 3.5 g/t Au and 23.0 g/t Ag over 2.0 m from a broader interval of 1.30 g/t Au and 12.98 g/t Ag over 6.03 m.

The best intercept from the L Zone was 2.18 g/t Au and 5.84 g/t Ag over 5.59 m from SR21-099, which tested its southeastern extent where very limited drilling has been completed. This hole was a 40 m step-out along strike from 1984 Drillhole 602.

10.3.4.6 Between the M Zone and Creek Fault

In addition to the well-defined zones at the Scottie Gold Mine, near-surface veins to the north of the M Zone and to the south of Creek Fault were intersected by Drillholes SR21-095, SR21-097, SR21-099, SR21-102, SR21-103, SR21-106, SR21-106, SR21-109, SR21-113, SR21-114, SR21-122, and SR21-124. Sulphidized shear veins are parallel to the M Zone and returned a few anomalous gold intercepts, however most results were sub-economic.

One exception is to the northeast of the M Zone where SR21-099 intersected multiple veins thought to be isolated from the currently defined M Zone, with assay results of 4.25 g/t Au and 2.0 g/t Ag over 2.0 m from a broader intercept of 1.07 g/t Au and 1.54 g/t Ag over 13.0 m. To the northwest of the M Zone, the best result was 0.85 g/t Au and 16.65 g/t Ag over 2.82 m from SR21-097.

10.3.5 Domino

Thirteen holes were drilled in the Domino area in 2021 for a total of 2025.8 m. Drilling in the main Domino area, located in the eastern part of the zone, was designed to follow up on 2020 drilling. Similarly to 2020, drilling returned numerous intervals greater than 1 g/t Au, however mineralized intercepts are narrow in width.

The highest-grade result from the 2021 drilling at Domino was from SR21-074 with 31.4 g/t Au and 129 g/t Ag over 0.37m.

To the west of the main area, holes SR21-101, SR21-104, SR21-107, and SR21-110 were drilled at the Gloria area. Never drill-tested, the Gloria showing hosts sheared pyrrhotite-rich quartz-carbonate veins resembling those found at the Scottie Gold Mine area. Rock sampling of these vein in 2020 returned up to 81.7 g/t Au and This showing is also underlain by a coincident conductivity high, resistivity low anomaly identified in a 2020 IP geophysical survey. The best intercept from Gloria was from SR21-104 of 4.89 g/t Au and 6.64 g/t Ag over 2 m.

10.4 2023 Drilling

The 2023 exploration at the Scottie Gold Mine project included: (1) 20,129.5 m of diamond drilling focused on the Stockwork Zone, Blueberry Contact Zone, Gulley Zone, Lakebed, C Zone, D Zone, E Zone, and P Zone and; (2) Property wide prospecting and sampling.

Eighty-four diamond drillholes, totaling 20,129.5 m, were completed on the Scottie Gold Property. The drilling was focused on Stockwork, Blueberry Contact, Gulley, Lakebed, C Zone, D Zone, E Zone, and P Zone.

10.4.1 Stockwork zone

Three holes for a total of 1,018.3 m were drilled at the Stockwork zone. Two holes, SR23-294, SR23-295, intersected intervals of anomalous gold and associated pervasive weak to moderate silica and sericite alteration. SR23-294 intersected broad, elevated Au intervals including 0.43 g/t Au over 75 m. SR23-295 intersected broad, anomalous Au throughout the hole with 0.33 g/t Au over 25 m and 1.47 g/t over 4.65 m. SR23-299 intersected intervals of weak to moderate Quartz-Sericite-Pyrite alteration and weak Au mineralization including intervals of 1.04 g/t Au over 2 m and 1.1 g/t Au over 4 m.

10.4.2 Bend Vein

The Bend Vein was targeted in two drillholes for a total of 202 m. The primary objective of drilling at the Bend Vein was to extend mineralization along strike to the west from existing mineralization. Neither hole intersected significant Au values. Notable assays returned 18 g/t Ag, 2394 g/t Pb and 3935 g/t Zn over 1 m from SR23-290 and 28 g/t Ag over 2 m from SR23-292.

10.4.3 Blueberry Contact

A total of 15,590.5 m comprising 61 holes were drilled to test the Blueberry Contact zone and associated Road, Blueberry, Fifi, Lemoffe, Gulley Zone, and E zones. Mineralized zones with >0.5 g/t Au were intersected in 55 of the 61 drillholes, with 49 drillholes returning assay values greater than 1 g/t Au in one or more intervals. Silver values greater than 30 g/t were intersected in 34 of 61 holes.

10.4.3.1 Road

The Road Zone was targeted in drillholes SR23-247, SR23-249, SR23-251, SR23-256, SR23-265, SR23-278, and SR23-281. Mineralization is spatially related to intercepts of altered and/or deformed rocks. Notable intercepts from the Road Zone include 4.6 g/t Au and 3 g/t Ag over 1 m from SR23-247, 6.71 g/t Au and 29 g/t Ag, 19.5 g/t Au and 3g/t Ag over 1 m intervals from SR23-249, and 7.7 g/t Au and 4.5 g/t Ag over 2 m from SR23-261. A shallow, Cu-Ag-bearing interval was intersected in SR23-261 with 230 g/t Ag and 1.3% Cu over 1 m at a true depth of 20 m.

10.4.3.2 Blueberry Vein

The Blueberry Vein zone and its intersection with the Blueberry Contact zone were targeted in drillholes SR23-241, SR23-247, SR23-249, SR23-255, SR23-256, SR23-261, SR23-264, SR23-265, SR23-268, SR23- 273, SR23-274, SR23-275, SR23-276, SR23-279, SR23-284, SR23-311, SR23-313, and SR23-315. Results from the Blueberry Vein structural corridor was variable, with the best results returned from the northern part of the structure underlying the Road Zone, which include intercepts of 13.87 g/t Au over 2.04 m from S R23-247 and 26.9 g/t Au and 2 3.8 g/t A g over 4 m. Auriferous mineralization associated with the Blueberry Vein has been extended to the north by ~105 m from previous drilling results with an intercept of 26.7 g/t over 1 m and 14.5 g/t over 3.7 m in SR23-261. SR23-247 returned 13.87 g/t Au and 3.94 g/t Ag over 2.04 m and 26.9 g/t Au and 23.75 g/t Ag over 4 m.

10.4.3.3 Fifi Vein

Drillholes that intercepted the Fifi Vein zone are SR23-235, SR23-236, SR23-238, SR23-242, SR23-254, SR23-255, SR23-259, SR23-264, SR23-268, SR23-271, SR23-273, SR23-275, SR23-276, SR23-279, SR23- 284, SR23-306, SR23-308, SR23-309, SR23-310, SR23-311, SR23-312, SR23-313, and SR23-314. Notable intercepts from the Fifi Vein zone are 19.5 g/t Au and 14.7 g/t Ag over 12.7 m from SR23-236 including 99.5 g/t Au and 51.5 g/t Ag over 2 m and 7.45 g/t Au and 7.1 g/t Ag from SR23-279 over 14.2 m including 36.1 g/t Au and 26.9 g/t Ag over 2.3 m. The deepest Au bearing intercept of the Fifi Vein is 2.6 g/t Au and 3.5 g/t Ag from SR23-256 at a true depth of approximately 395 m.

10.4.3.4 Lemoffe Vein

Drillholes targeting the Lemoffe Vein zone are SR23-235, SR22-236, SR22-238, SR22-239, SR22-242, SR22-245, SR22-254, SR22-255, SR22-256, SR22-259, SR22-264, SR22-265, SR22-268, SR23-271, SR23-274, SR23-276, SR23-284, SR23- 306, SR23-308, SR23-309, and SR23-310. Notable intercepts in this zone are 88.4 g/t Au and 25 g/t Ag over 2 m from SR23-264, 28.15 g/t Au and 17.5 g/t Ag over 4 m from SR23-268, and 59.2 g/t Au and 16 g/t Ag over 1.25 m from SR23-306. The deepest Au bearing intercepts in the 2023 drill program have been interpreted to be the intersection of the Blueberry Contact Zone and the lower extent of the Lemoffe Vein zone with 3.53 g/t Au, 43 g/t Ag, 6528 g/t Pb and 7381 g/t Zn over 1.04 m from SR23-235 and 6.5 g/t Au over 1 m in SR23-265 with both intercepts at a true depth of approximately 550 m. The highest Zn values observed from the 2023 program were interpreted to be from the Lemoffe Vein zone in SR23-242 with 3.24% Zn (0.07 g/t Au and 88.8 g/t Ag) over 2.18 m and 3.8% Zn (0.36 g/t Au and 15 g/t Ag) over 1.38 m from SR23-242.

10.4.3.5 Gulley

Twelve holes were drilled in the Gulley zone for a total of 2,455.5 m. The holes were designed to test Blueberry Contact Zone-style mineralization in the Gulley zone. Drillholes SR23-234, SR23-257, SR23-258, SR23-260, SR23-272, SR23-280, SR23-282, SR23-285, SR23-296, SR23-305 and SR23-307 targeting the Blueberry Contact zone intercepted strongly sulphidized (pyrrhotite-pyrite-arsenopyrite-galena- sphalerite) andesite and siltstone associated with moderate to intense sericite-chlorite alteration. The highest Ag-bearing intercepts of the 2023 program were encountered in the Gulley zone. Assays returned 1.2 g/t Au and 356.1 g/t Ag over 5.35 m, and 3.7 g/t Au and 78 g/t Ag over 7.45 m from SR23-234, 1.5 g/t Au and 14.6 g/t Ag from SR23-305, and 8.13 g/t Au and 11 g/t Ag over 1 m from SR23-307. The interval with the highest Cu value from the 2023 drill program was from SR23-285 with 3.02% Cu and 314 g/t Ag over 1 m at a true depth of 85 m.

10.4.3.6 E Zone

Eight holes targeted at the E Zone for a total of 1,844.7 m. Drillholes SR23-237, SR23-243, SR23-262, SR23-263, SR23- 266, SR23-267, SR23-269, and SR23-270 were designed to test geophysical targets and to follow up on significant pyrrhotite-pyrite-arsenopyrite-galena-sphalerite mineralization, and minor chlorite-sericite-carbonate alteration intercepted in SR22-227. Notable intervals include 1.38 g/t Au and 15.7 g/t Ag over 3.2 m from SR23-237, 2.26 g/t Au and 94.4 g/t Ag over 3.2 m from SR23-263, 1.62 g/t Au and 10.1 g/t Ag over 3.3 m from SR23-267. Moderate base metal intervals were intercepted in the E Zone including 2.27% Zn over 3 m in SR23-263 and 1.36% Zn over 2 m in SR23-270.

10.4.4 C Zone

Three holes were drilled at the C zone for a total of 471.8 m. The holes were designed to follow up on surface work and drilling results from previous campaigns. Gold-bearing, mineralized veins, and shear structures were intersected in two holes in the western extent of the C Zone during the 2022 drill program. The C Zone structures are characterized by pyrrhotite-pyrite ± chalcopyrite-arsenopyrite-sphalerite-galena with associated quartz, carbonate, and sericite-chlorite 31 alteration. SR23-300, SR23- 301, SR23-302 intersected zones of moderate sericite and chlorite alteration with overall weak sulphide mineralization. Assays returned 2.58 g/t Au and 10 g/t Ag over 2 m from SR23-300.

10.4.5 D Zone

Five holes were drilled at the D zone for a total of 1,299.6 m. Drilling in 2023 targeted mineralized veins hosting pyrrhotite-pyrite±chalcopyrite±sphalerite±galena with associated moderate to intense silica and sericite-chlorite alteration. Gold ± silver-bearing sulphide veins were intercepted in all five of the holes (SR23-286, SR23-287, SR23-289, SR23-303 and SR23-304). Notable assays returned 36.3 g/t Au and 37.5 g/t Ag over 5 m from SR23-286, and 20.1 g/t Au and 30 g/t Ag over 1 m from SR23-289.

10.4.6 Scottie Gold Mine – P Zone

Drilling at the Scottie Gold Mine consisted of two holes for a total of 407.4 m. The main objective of drilling at the Scottie Gold Mine was to expand the mineralized extent of the P Zone along strike and at depth. Both holes intercepted Au-bearing sulphide mineralization and moderate sericite-chlorite alteration. Notable assay results include 6.89 g/t Au and 4.2 g/t Ag over 5.8 m from SR23-293, 7.9 g/t Au and 2 g/t Ag over 2 m, 7.11 g/t Au and 25.4 g/t Ag over 1.65 m and 2.17 g/t Au and 3 g/t Ag over 8.85 m from SR23-298.

10.4.7 X Zone

A single hole, SR23-297, was to test a 2022 HLEM anomaly which lies beneath the Morris Summit Fault. Drilling did not produce significant results with a single intercept of 2.31 g/t Au and 104 g/t Ag over 1.05 m from SR23-297.

10.4.8 Lakebed

Nine holes were drilled in the Lakebed zone to test mineralization along the siltstone-andesite contact to the south of the Blueberry Contact Zone. Drillholes SR23-240, SR23-243, SR23-244, SR23-246, SR23- 248, SR23-250, SR23-253, SR23-283, and SR23-288 did not produce significant results. The best intercepts were from SR23-240 located to the east of 32 the Gulley zone and included 1.22 g/t Au and 3 g/t Ag over 2.75 m, and 0.93 g/t Au over from SR23-248 located ~420 m southeast of the E Zone.

10.5 2024 Drilling

Forty-four diamond drillholes, totalling 10,270 m, were completed on the Scottie Gold Property (Figure 6). Drilling commenced on the 16th of July and finished on the 13th of September. Driftwood Diamond Drilling Ltd. was contracted to complete the programme, using two diamond drills.

The Blueberry Contact Zone is road accessible; thus, skid drills were used to test the planned targets in the zone. D Zone, Golden Buckle Zone and Scottie Gold Mine targets required pad construction and helicopter assistance to complete the drilling programme. All technical teams were stationed at the Granduc camp from where they accessed both skid and drill pads daily.

10.5.1 Blueberry Contact Zone

A total of 4,612 m of drilling, across 18 holes, was completed to test the Blueberry Contact Zone and its associated Road, Blueberry, Fifi, Lemoffe, Gulley, and Serac zones. Fourteen of the eighteen holes returned intervals grading greater than 1 g/t Au.

10.5.1.1 Road Zone

The Road Zone was tested by Drillholes SR23-317, SR24-318, SR24-320 and SR24-359. These holes were designed to infill a gap and extend mineralization between Road Zone and Road West Zone. Notable intercepts include: 6.6 g/t Au over 4 m, including 22.8 g/t over 1 m in hole SR24-317, 14.7 g/t Au over 4 m, including 52.6 g/t over 1 m in hole SR24-359 and 12.0 g/t Au over 2.7 m including 22.7 g/t in hole SR24-320. Mineralization is dominated by vein-hosted and disseminated pyrite, pyrrhotite and arsenopyrite. It is hosted in altered siltstones which are strongly silicified and overprinted by sericite-chlorite and carbonate alteration.

10.5.1.2 Blueberry Vein

The Blueberry Vein zone and its intersection with the Blueberry Contact Zone were targeted in Drillholes SR24-317, SR24-355, SR24-356, SR24-357, SR24-358 and SR24-359. A notable intercept from the Blueberry Vein Zone includes 10.8 g/t Au over 3.0 m including 21.0 g/t Au over 1.0 m in hole SR24-359. Mineralization is associated with chlorite and sericite alteration, particularly within faulted and sheared intervals. Vein-hosted mineralization is prominent and locally massive, commonly accompanied by pervasive dark chlorite alteration.

10.5.1.3 Fifi Vein

Drillholes that intercepted the Fifi Vein zone are SR24-321, SR24-355, SR24-356, SR24-357, SR24-358 and SR24-359. The latter drillholes targeting the Fifi Vein Zone were proposed as infill targets for the Blueberry maiden resource. The two highest Au intercepts from the Blueberry Contact zone were from the Fifi veins in holes SR24-355 and SR24-357. Hole SR24-355 intercepted 30.2 g/t Au over 5 m from a depth of 105.90 m including 75 g/t over 1.45 m and 37.3 g/t over 1.05 m. SR24-357 intercepted 47.40 g/t over 2 m with 68.30 g/t over 1 m.

10.5.1.4 Lemoffe Vein

Drillholes that intercepted the Lemoffe vein zone are SR24-321, SR24-355, SR24-356, SR24-357, SR24- 358 and SR24-359. The Lemoffe vein intercepted high-grade Au intervals with the highest grade coming from SR24-321 with grades of 12.30 g/t over 3.27 m with 24.2 g/t over 1.2 m. And SR24-356 returned grades of up to 35.2 g/t over 2.0 m with 39.1 g/t over 1.0 m. The holes intercepted a thick andesite unit and strongly altered and sheared units characterized by mineral abundance. Alteration assemblages consist of chlorite-sericite alteration and silicification. Altered units host polymetallic multiphase veins and exhibit a weak-moderate foliation.

10.5.1.5 Gulley Zone

Drilling at the Gulley zone aimed to expand the mineralization to the north in holes SR24-325, SR24-346 and SR24-350. Multiple holes returned intervals >1 g/t Au. Hole SR24-325 returned the highest intercepts with grades of 4.22 g/t Au over 1 m, 3.4 g/t Au over 2 m, and 4.91 g/t over 1 m.

10.5.1.6 Serac Vein

Drilling at the Serac zone was designed to determine the orientation and test the southern extension of the Serac vein with holes SR24-319, SR24-323, SR24-352 and SR24-354. The holes intercepted multiple large polymetallic, polyphase veins as well as hydrothermal breccia. However, samples intercepting the Serac vein returned relatively low grade. The highest overall intercept, from SR24-352, returned up to 2.1 g/t Au over 3 m in an altered andesite.

10.5.2 D Zone

Three holes, comprising 830 m, targeted the D zone in the 2024 season. SR24-324, SR24-326 and SR24-328 were designed to determine the orientation and thickness of D veins. The D vein was intercepted at about a depth of 243 m, comprising massive pyrrhotite stringers, vein hosted pyrite with trace chalcopyrite and sphalerite. Noticeable intercepts include 7.1 g/t Au over 1 m in SR24-324 and 5.5 g/t over 1.1 m in SR24-326.

10.5.3 Golden Buckle Zone

Three drillholes totaling 433 m were completed at the Golden Buckle Zone during the 2024 season. This marked the first drill testing of the Steep Zone. Holes SR24-348, SR24-349, and SR24-351 were designed to follow up on high-grade rock samples (>20 g/t Au) collected in 2023. Two of the three holes intersected gold grades exceeding 1 g/t Au.

The drill core is composed predominantly of unmineralized to weakly mineralized andesite, including intervals of sheared andesite with localized mineralization. Alteration in the host andesite is characterized by moderate pervasive to strong patchy chlorite alteration, with strong sericite-silica and moderate to strong potassium feldspar alteration occurring in halos around, or spatially associated with, quartz-dominant veins. Mineralization is primarily composed of trace pyrrhotite and pyrite, with local occurrences of vein-hosted sphalerite, chalcopyrite, and molybdenite. Notable intercepts include 1.02 g/t Au over 1.05 m in hole SR24-348 and 1.87 g/t Au over 1.00 m in hole SR24-349.

10.5.4 Scottie Gold Mine (SGM)

Twenty drillholes, totaling 4,395 m, were completed at the Scottie Gold Mine, targeting the L, M, N, O, and P zones, as well as the newly discovered Wolf Zone. The primary objectives of the drilling were to confirm grades and collect data necessary for a maiden resource estimate, test mineralization in the footwall of the Morris Fault, and expand known mineralization in the targeted zones.

Mineralization intersected at the Scottie Gold Mine is predominantly hosted within altered and sheared andesitic rocks and associated polymetallic veins. Sulphide mineralization ranges in abundance from 5% to 80% and is primarily composed of pyrrhotite and pyrite.

10.5.4.1 P Zone

The P Zone was targeted by Drillholes SR23-336, SR24-337, SR24-338 and SR24-340. The holes were designed to test the downdip structures and extend mineralization in P Zone and M Zone. Gold mineralization was intercepted in three holes, successfully expanding mineralization to the northwest and southwest. Hole SR24-337 intersected multiple high-grade intercepts with 9.5 g/t Au over 4 m including 54 g/t and 4.9 g/t over 2 m with 6.48 g/t over 1 m.

10.5.4.2 O Zone

The O Zone was targeted in Drillholes SR24-330, SR24-332, SR24-334, SR24-336, SR24-342, SR24-343, SR24-345 and SR24-347. The 2024 drilling was successful at the O zone, expanding known mineralization to the east with three high-grade, near surface intercepts. Holes SR24-330, SR24-332, and SR24-334 respectively intersected 4.5 g/t over 12.5 m, 6.4 g/t over 10.7 m and 10.2 g/t over 8.95 m. Highlighted high-grade interests occurred at depths lesser than 25 m.

10.5.4.3 M Zone

The M Zone was targeted by Drillholes SR24-327, SR24-329, SR24-330, SR24-331, SR24-332, SR24-333, SR24-334, SR24-335, SR24-339, SR24-341, SR24-344 and SR24-353. Notable intercepts from the M Zone include: 26.1 g/t Au over 2.0 m, including 43.7 g/t over 1.0 m in SR24-327, 13.5 g/t Au over 2.0 m, including 22.0 g/t over 1.0 m in SR24-330, 23.4 g/t Au over 1.1 m in SR24-334, 9.5 g/t Au over 3.0 m in SR24-339 and 12.2 g/t Au over 1.0 m in SR24-344.

10.5.4.4 N Zone

The N Zone was targeted in Drillholes SR24-329, SR24-330, SR24-331, SR24-332, SR24-333, SR24-334, SR24-339, SR24-341, SR24-344 and SR24-353. The 2024 drill program successfully extended mineralization along strike to the south-east in SR24-330 with 13.48 g/t Au over 2.0 m and 8.76 g/t over 6.1 m including 10.1 g/t Au over 1.21 m and 22.0 g/t over 1.0 m. Other notable intercepts at the N zone are 6.52 g/t over 4.19 m with 19.6 g/t over 1.0 m in SR24-333, 6.54 g/t over 5.5 with 10.2 g/t over 1.5 m in SR24-334, and 49.4 g/t over 3.1 m with 83.7 g/t over 1.0 m and 48.4 g/t over 1.1m.

10.5.4.5 L Zone

The L Zone was targeted in Drillholes SR24-334, SR24-353. There were no intercepts greater than 1.0 g/t from the L zone. The highest intercepted grade overall was 0.49 g/t Au over 3 m from SR24-334.

10.5.4.6 Wolf Zone

The Wolf Zone was drilled for the first time in 2024, following up on a high-grade surficial sample Y610817 which yielded 4.04 g/t Au. The sample was collected in 2019 adjacent to the retreating glacier. Hole SR24-353 intercepted multiple polymetallic shear veins that returned 6.5 g/t Au over 2.15 m including 11.9 g/t over 1.0 m from a depth of 289.50 m and 19.4 g/t over 2.0 m including 24.4 g/t over 1.0 m from a depth of 313.50 m.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Preparation and Security Scottie Resources

11.1.1 Procedures for 2021 to 2024

In 2024, a total of 5,397 core samples were taken with an average length of 1.87 m, ranging from 1.0 to 3.0 m. The hole was sampled from top to bottom. Core samples were cut in half with an electric core cutting saw, then either processed at a local mobile prep unit in Stewart, BC, or shipped to the SGS Canada ("SGS") preparation lab in Burnaby, BC, via Scottie Resources, Serac Exploration and Bandstra Transportation (Smithers, BC). The remaining half core is stored at the Bitter Creek Staging area off of Hwy 37A.

An additional 747 QAQC samples were inserted (317 blanks, 306 CRMs, 124 coarse duplicates) for an insertion rate of 13.8% that meets industry best practice (e.g. Abzalov, 2008). Field duplicate pairs were quartered with the two quarters submitted for analysis, leaving half of the core in the core box.

11.1.2 Procedures from 2016 – 2020

The following drilling and sampling procedures for the 2016 to 2020 drilling campaigns were compiled by Equity Exploration Consultants, with minor edits by MMTS:

In 2020, a total of 3,594 core samples were taken with an average length of 1.32 m and range of 0.3 to

3.1 m, for 67% of all drill core sampled. Core samples were split and bagged in the same manner as the 2019 work, then shipped to the MS L BS Inc ("MS ") preparation lab in Terrace, BC, via Scottie Gold, Equity, and Rugged Edge Holdings Ltd (of Smithers, BC). The remaining half core is stored at the Bitter Creek Staging area off Hwy 37A.

An additional 501 QAQC samples were inserted (168 blanks, 166 CRMs, 84 coarse duplicates, 83 field duplicates) for an insertion rate of 12% that meets industry best practice (e.g. Abzalov, 2008). Field duplicate pairs were quartered with the two quarters submitted for analysis, leaving half of the core in the core box.

The 2019 and 2020 drill programs were both managed by Equity on behalf of Scottie Gold. In 2019, a total of 866 core samples were taken with an average length of 1.91 m and range of 0.3 to 3.0 m, for 84% of all drill core sampled. A core saw was used to split the core along the apical line, with half the sample submitted for analysis and the other half left in the core box for reference. The remaining half core is stored on the concrete pad at the Scottie Gold Mine historical camp site.

Cut samples were placed in a poly-ethylene bag along with a barcoded sample tag, then zip-tied, bundled into rice bags, sealed with a numbered security tag, and shipped to the ALS Geochemistry (" LS ") preparation lab in Terrace, BC, by Equity and Scottie Gold personnel.

An additional 137 QAQC samples were inserted (63 blanks, 49 CRM, 25 coarse duplicates) for an insertion rate of 14% that meets industry best practice (e.g. Abzalov, 2008). Field duplicate pairs were quartered with the two quarters submitted for analysis, leaving half of the core in the core box.

The 2016 drill program was managed by Rotation and included the taking of 692 half core samples with an average length of 1.3 m in a range of 0.3 to 3.4 m. A total of 858 m of core was sampled out of 2,648 m drilled, for 26% of all metres drilled. The remaining half core is stored at Roland Soucie's storage yard a few kilometres north of Stewart.

One certified reference material (CRM) and one blank were inserted with every 18 core samples, for a quality control (QC) insertion rate of 10%. No duplicate samples were taken. Available documents do not provide descriptions of core processing procedures, such as core cutting, sample registration, and chain of custody.

11.1.3 Sample Preparation and Security Historical

Details about the 2016 sample preparation and security as well as the QA/QC protocols were not reported (ARIS report 36674 by Rotation Minerals Ltd.). The core from the drilling program is being stored at 426 King Street, Stewart, in core storage facilities owned by Decade Resources Ltd. According to the data made available to MMTS, the insertion rates for blind standards are 3.8%, for blanks 3.9%, and for coarse and pulp duplicates combined 12.7%. No field duplicates were taken in 2016.

Protocols for drilling and sampling in 2006 are not available, but it appears that from Drillholes 737-762, no quality control samples were added to the sample stream at all. For Drillholes S06-01 to S06-04 at Blueberry, however, blanks and blind standards were inserted at regular intervals of one every 20 samples each. MMTS has no records of precision control of any kind for 2006.

Sampling, sample preparation and chain of custody details for the 2004-2005 drill seasons were compiled by Gunning et al (2006), as follows with minor text adjustments:

Drill core was split in half at Summit Lake using a rock saw splitter. One half of the core was stored in a sample bag with the remaining half being returned to the core box. Sample length was determined based on obvious mineralization, veining and alteration. Efforts were made throughout the program not to enter any biases into the splitting. Core recovery through the mineralized zones was excellent with losses being minimal. A prepared blind standard and a blank were entered into the sample stream at 20 sample intervals.

All samples were packed into plastic bags, sealed then placed into rice bags and the bags tied off. The bags were then trucked by company personnel to Eco- Tech Lab/s prep lab in Stewart where the samples were crushed and pulverized with the resulting pulp being forwarded to Eco-Tech Labs, 10041 East Trans Canada Highway, Kamloops, BC for analysis. No aspect of the sample preparation was conducted by an employee, officer, director or associate of the issuer.

MMTS is not aware of any custom QA/QC data generated prior to 2004. Copies of original certificates are scarce, so the lab-internal quality control data could not be reviewed in any meaningful quantity.

11.2 Laboratory Procedures by Year-Lab

Several of the following lab procedure descriptions were copied from the 2021 Technical Report on the Scottie Gold Mine by Equity Exploration Consultants and subsequently updated.

11.2.1 MSA 2020

MSA is independent of Scottie Gold and is accredited by the International Accreditation Service (IAS) as having demonstrated compliance with ISO/IEC 17025:2017. MSA is not accredited by the Standards Council of Canada. The IAS certifies that MSA is accredited to complete the analytical methods requested by Scottie Gold, including the determination of gold by lead collection fire assay and atomic absorption spectrometry (FAS-111), gold and silver by lead collection fire assay and gravimetric finish when high-grade (FAS-413/FAS-415 for >10g/t Au and FA-418 for >100g/t Ag), and multiple elements including Ag by four-acid digestion and ICP-ES/MS finish (IMS-230). The detection limits are 0.005g/t and 0.01g/t for Au and Ag, respectively.

Samples received at the MS Analytical Terrace preparation facility were crushed to 70% passing 2mm. A 250 g riffle split was taken and pulverized to 85% passing 75µm. The resulting pulps were then shipped to Langley, BC, for fusion, digestion, and analysis at MS ’s main lab facility.

In addition to the two main analysis methods FAS-111 and IMS-230, several base metal ‘ore grade’ methods were triggered on occasion when the initial analysis reported >10,000ppm of either Cu, Pb, or Zn (ICF-6 code which involves a 0.2g sample digested by a 4-acid mix and analysis by ICP-AES). Samples containing cobalt exceeding 1,000ppm were re-analyzed by PER-7Co (sodium peroxide fusion of a 0.15g sample cut with ICP-AES finish and a lower report limit of 20ppm).

Select samples were also analyzed for ‘whole rock’ composition using method WRX-310 which entails a lithium borate fusion of 0.6g of sample material with XRF analysis.

11.2.2 SGS 2021-2024

SGS is independent of Scottie Gold and is accredited by the International Accreditation Service (IAS) as having demonstrated compliance with ISO/IEC 17025:2017.

Logged and cut drill core samples were dried, weighted, and crushed to approx. 70% passing 2mm. A 250g split was taken to a Cr steel pulverizer to reduce the grain size of the sample to 85% passing 75µm. This was done either in an SGS-owned MSPU in Stewart, BC, (2021-2024) or at the SGS preparation facilities in Burnaby, BC, for most of the 2024 samples.

The resulting pulps were then transported to and analyzed at SGS in Burnaby, BC, as per the following methods:

For Au, method GE_FAA30V5 was requested which is a fire-assay procedure on a 30g sample with AAS finish and a lower detection limit of 5ppb. Exceedance of the upper reporting limit of 10,000ppb triggered an additional ‘ore grade’ fire assay analysis with gravimetric finish (GO_FAG30V) with a reporting window of 0.5 to 10,000g/t.

Some metallic screening data was produced on select high-grade samples (GO_FAS30M).

For Ag and a suite of 32 or 48 other elements, depending on selected finishing instrumentation, all samples were broken down using a 4-acid mix on 0.2g of material for a ‘near-total’ digestion. o r analysis GE_ICP40Q12 utilizes an ICP-OES exclusively while GE_IMS40Q12 involves both the ICP-OES setup and an ICP-MS for very low detection limits for certain metals (including Ag). GE_IMS40Q12 was requested for approx. 3,000 out of the total of approx. 39,000 SGS-analyzed samples (including QA/QC) in 2021 and 2022.

‘ re -grade’ g concentrations were determined using method G _ICP 42Q100 for samples that exceeded the initial 100g/t upper reporting limit. The digestion procedure is the same but with higher H2O dilution before ICP-OES analysis. The reporting limits of the method is 100g/t to 1,000g/t Ag.

11.2.3 ALS 2018 – 2021

ALS is independent of Scottie Gold, accredited under the Standards Council of Canada testing and calibration laboratory accreditation program (LAP, lab no. 579), and meets the General Requirements for the Competence of Testing and Calibration Laboratories (ISO/IEC 17025:2017) as defined by the International Organization for Standardization (ISO). Under LAP, ALS is certified to complete the analytical methods requested by Scottie Gold, including the determination of gold by industry-standard lead collection fire assay and atomic absorption spectrometry (Au-AA23), Au over-limits by the same fire-assay procedure plus gravimetric finish (Au-GRA21) and multiple element determination by aqua regia digestion and ICP-AES finish (ME-ICP41).

Samples received at the Terrace preparation facility were crushed to 70% passing 2 mm. A 250 g riffle split was taken and pulverized to 85% passing 75 µm before sending the pulps to North Vancouver for geochemical analysis. Standard fire-assay method Au-AA23 processes a 30g sample and reports a lower detection limit of 0.005g/t. In case of exceedance of the upper reporting limit of 10g/t, a second 30g cut gets processed and analyzed using Au-GRA21 with gravimetric finish.

As for the multi-element side of the available data including Ag, the 2019 samples were digested using an aqua regia solution (ME-ICP41) with ICP- E S finish, with the occasional ‘ore grade’ analysis of Cu or Zn where the upper reporting limit was exceeded (ME-OG46). For some limited additional shoulder sampling of 2020 core and all 2021 drill core that was sent to ALS (samples from 10 holes only), the more comprehensive 4-acid ME-ICP61 method was requested.

In 2018, some limited gap sampling (on 2004, 2005, 2006, and 2016 core) and check-assay selecting was completed (2 certificates with 125 total samples), the analyses of which were done by ALS. In contrast to previous and future analysis methods at ALS, Scottie Gold chose Au-ICP21 for Au and ME-MS61 for all other elements. Au-ICP21 is a 30g fire-assay method with ICP-AES finish while ME-MS61 is a 4-acid digest of 0.25g of sample material with an ICP-MS finish and very low reporting limits for most metals.

11.2.4 Loring Labs 2016

Loring is independent of Scottie Gold, accredited under the Standards Council of Canada testing and calibration laboratory accreditation program (LAP, lab no. 868), and meets the General Requirements for the Competence of Testing and Calibration Laboratories (ISO/IEC 17025:2017) as defined by the International Organization for Standardization (ISO). Under LAP, however, Loring lacked the certification to complete the analyses that were requested by Rotation Minerals at the time, including the determination of gold by lead collection fire assay and atomic absorption spectrometry on a 30g sample. Typical overlimit methods for gold (e.g. gravimetric, screen assay) were apparently not requested for the 2016 analyses. However, according to the original lab certificates, samples that returned >10,000ppb on the initial Au assay were re-analyzed with a more suitable but currently unknown method of higher detection limit and reported in g/t.

The equally available 30 element suite including Ag are the product of a hot aqua regia digestion with ICP finish. The Ag lower detection limit is 0.5g/t.

11.3 Laboratory Procedures Historical

11.3.1 EcoTech 2004 – 2005

According to Gunning et al (2006), Eco-Tech was an ISO-9001 accredited laboratory at that time. At Eco- Tech the samples are sorted, dried (if necessary) then crushed through a jaw crusher and cone or roll crushed to –10 mesh. The sample is then split through a Jones riffle until a 250g sub sample is achieved. The sub sample is pulverized in a ring & puck pulverizer to 95% passing 140 mesh. The sample is then rolled to homogenize.

For gold analysis a 30 g sample size is fire assayed using appropriate fluxes. The resultant doré bead is parted and then digested with aqua regia and then analyzed on a Perkin Elmer AA instrument. For the 30 element ICP suite a 0.5g sample is digested with 3 ml of 3:1:3 nitric acid to hydrochloric acid to water at 90 degrees for 1.5 hours. The sample is then diluted to 20 ml with demineralized water and analyzed using a Jarrel Ash Inductively Coupled Plasma Analyzer. At Eco-Tech every 20th sample is an in-house standard and every 40th sample is duplicated. This is undertaken to control accuracy and reproducibility of the laboratory analysis. In addition, a second (coarse) split is taken every 35 samples to determine the homogeneity of the mineralization.

11.3.2 1983 – 1991

In the Technical Report for the Summit Lake Property (2004), Visagie lists previous labs and procedures as follows:

The 1991 drill core samples taken at the Bend Zone were analyzed at the Premier Mine lab operated by Westmin Resources using a regular fire-assay method with atomic absorption finish (FA-AA).

In 1990, drill core was analyzed by Eco-Tech in Kamloops, after sample prep was completed in Stewart. Samples were dried and crushed, then pulverized to -140 mesh. FA-AA on a 1 assay ton (AT) sample was used for Au and an aqua regia digestion with atomic absorption finish for Ag.

A 1987 drill core was processed at the Newhawk Goldmines laboratory, using fire assay with gravimetric finish on 1AT. Total samples were crushed to –1/4 inch and a 250 to 500g sample split off in a Jones Riffle Splitter and sent to the pulverizer. The sample is pulverized to a fine powder (no screen analyses available), rolled on a sample canvas and a one assay ton taken for fire assay. Classic fire assay methods with gravimetric finish were used to provide assays for gold and silver. The method involves firing with litharge to produce a lead pellet, refiring to remove the lead and provide an Au-Ag bead, weighing this bead, dissolving the Ag with nitric acid, refiring to anneal the Au bead and weighing the gold bead. The scales are considered accurate to +/- 0.002 opt Au.

In 1983, samples were either processed at the Scottie Goldmine lab in Stewart, BC, or the Min-En-operated Premier Mine lab 20km up the road. t the Scottie lab, samples are crushed to ¼" then split down to approximately 200 grams. The rejects are discarded. The 200g split is pulverized and the sample assayed using a ½ assay ton sample with the remaining pulp being stored. The ½ assay ton sample is fused using traditional fire assay techniques and the bead weighed.

t Premier, samples are crushed to ¼" then split down to 200 grams. The 200g split was then pulverized and a one-ton sample taken for fire assay. A silver pellet is added to assist in forming the bead during the fusing of the sample. The bead formed is then broken down with HBr and MIBk and solution analyzed by AAS for gold.

For check-assay purposes, sub-sets of pulps were sent to Vangeochem where the samples were fused in the normal fire assay procedure and the doré dissolved in dilute H2NO3 to eliminate the silver. The cinder is then dissolved in NH4OH and the solutions analyzed for gold by AAS.

11.3.3 QA/QC 2016 – 2024

This chapter summarizes and interprets the QA/QC information available for the years 2016 to 2024. Geochemical assay results were generated by 4 different labs during that period; hence the following sections will table and graph the data individually for each lab with a focus on Au. Accuracy control graphs expand into Ag as well.

Annual assessment reports including detailed QA/QC from 2018 onwards were compiled by Equity Exploration Consultants Ltd. (Equity). MMTS did review the relevant chapters or appendices of these reports for clarification and confirmation purposes and has copied certain background information, but the following tables and figures in Sections 11.5 and 11.6 are built almost exclusively from the sampling and assaying database provided by Scottie Gold, after validating the data and making corrections where appropriate. As such, total counts, insertion rates, and performance interpretations may differ slightly. In their reports, Equity highlighted several instances in which performance control led to failure detection, the definitions of which are basically equivalent to the ones used in this report, and subsequent data correction via re-assaying or similar.

As a general statement, Scottie Gold (and Rotation Minerals in 2016) followed industry-standard guidelines for frequency and overall distribution of quality control materials introduced into the sample stream. Table 11-1 shows the respective insertion rates per year. While both blanks and standards coverages consistently approach 5% of total core samples, the duplicates rates are more variable both as total duplicates and even more so as individual duplicate types.

Table 11-1: 2016 – 2024 QA/QC Insertions Summary

Year	Lab	Core Samples	Blanks	CRMs	Field Dup	Coarse Dup	Pulp Dup	QA/QC Total	% QA/QC	Check	Comment
2016	Loring	691	35	34	0	70	43	182	20.8%	0	
2018	ALS	19	0	0	0	0	0	0	0.0%	0	2016 gap samples
2019	ALS	866	63	49	0	25	0	137	13.7%	0	
2020	MSA	3,594	166	165	84	82	82	579	13.9%	0	

Year	Lab	Core Samples	Blanks	CRMs	Field Dup	Coarse Dup	Pulp Dup	QA/QC Total	% QA/QC	Check	Comment
2021	ALS	859	47	52	12	10	1	122	12.4%	0	With 2020 gap samples
2021	SGS	7,597	435	426	110	101	1	1,073	12.4%	78	MSA check assays
2022	SGS	12,077	688	692	271	0	0	1,651	12.0%	0	With 2020 gap samples
2023	SGS	9,778	561	554	224	0	0	1,339	12.0%	0	
2024	SGS	5,397	318	307	124	0	0	749	12.2%	0	
Total		**40,878**	**2,313**	**2,279**	**825**	**288**	**127**	**5,832**	**12.5%**	**78**	

11.3.4 Blanks

The Scottie assay database contains assay results for several different blank materials, both coarse and pre-prepped, that can generally be identified by name. MMTS discriminates the blanks further based on obvious geochemical signatures (mainly Ca, Mg, and Mn concentrations) to assess if the different materials vary in their capabilities of capturing potential cross-sample contamination. The discrimination results in the following list of blanks:

1. 'Blank' – used in 2016 and 2022, no description available
2. 'CDN-BL-10' – used mainly in 2022, a pre-prepped purchased blank made of granite
3. 'Coarse Blank Carb' – 2022-2024, consistently exceeding the UDL of Ca at 15%
4. 'Coarse Blank Qtz' – 2023-2024, contains hardly any Ca, Mg, or Mn
5. 'Coarse Blank Rock' – 2024, generally low Al, with variable Ca, Fe, Cu
6. 'EQ Blank' – 2019-2021, taken from a barren granitic intrusion (Equity 2021)

Table 11-2 summarizes the blank counts and failures plus failure rate by year. The 'blank' inserted in 2016 appear to perform poorest with an almost 30% failure rate, based on 10*DL for Au. Reviewing the preceding samples indicates that most, if not all, of the 'failures' are not caused by cross-sample contamination but rather natural background concentrations. On occasion, the blank results even exceed the results of the sample/samples before it. Due to the uncertainty about the material used, the 2016 contamination control is being viewed as unacceptable, and the data not graphed for this report.

There are no contamination concerns for the 2019-2021 ALS and the 2020 MSA sample preparation for which the 'EQ Blank' was used, since only one of 275 total assays passed the 5*DL warning line, and no failures are recorded (also not graphed for this report). In the later half of 2021, after switching primary labs from ALS to SGS, however, multiple blanks that were inserted purposefully directly following ultra high-grade intervals reported grades well past the failure threshold, with a maximum of 0.421g/t in blank B997198 of Drillhole SR21-133 at 55m that controlled sample B997197 which graded 24g/t (Figure 11-1). This proves that meaningful cross-sample contamination does occur when grades exceed 20g/t Au. The carry-over Au in these reviewed cases varies between 0.01% and 1.75%.

In addition to these targeted blanks, Scottie Gold inserted a second EQ blank immediately following the first EQ blank a few times. Those are still elevated enough to be classified as 10*DL failures in 2 out of 3 cases but show much lower Au concentrations (0.01 to 0.035g/t Au) which is broadly in line with host rock grades in proximity to mineralized structures.

In Drillhole SR22-144, SR22-145, and SR22-156, Scottie Gold inserted consecutive blanks after high-grade intervals five total times again, in this case the ‘Coarse blank Carb’ which was used extensively in 2022 and continued in 2023-2024 (Figure 11-2). Carry-over rates into the first blank are similar at 0.03% to 0.4% and in 2 cases the second blank is still significantly contaminated at 0.137 and 0.155g/t Au, respectively. In Drillhole SR22-275, which is contamination-controlled by ‘Coarse blank Qtz’ shown in Figure 11-3, two sets of 3 consecutive blanks were inserted following high-grade intervals. In both instances only the first blank is significantly contaminated at a rate of 3.2% and 0.18% of the Au concentration of the high-grade sample, respectively, and the second and third blanks came back nearly clean.

Overall, for the complete dataset, very few un-targeted blanks exceed the 10*DL failure threshold or even the 5*DL warning line for that matter.

MMTS concludes that contamination from high-grade core samples into following low-grade core samples during preparation is not likely to affect more than one sample in the stream. Contamination appears to depend on Au particle size more than total grade as the correlation between Au grade in core sample vs. Au grade in following blank is weak.

Table 11-2: 2016 – 2024 Blanks Summary

Year	Lab	Core Samples	BLK Count Au	Fails Au	% Fails Au
2016	Loring	691	35	10	28.6%
2018	ALS	19	0	0	0.0%
2019	ALS	866	63	0	0.0%
2020	MSA	3,594	166	0	0.0%
2021	ALS	859	47	0	0.0%
2021	SGS	7,597	435	5	1.1%
2022	SGS	12,077	688	8	1.2%
2023	SGS	9,778	318	5	1.6%
2024	SGS	5,397	254	2	0.8%
Total		**40,878**	**2,006**	**30**	**1.5%**

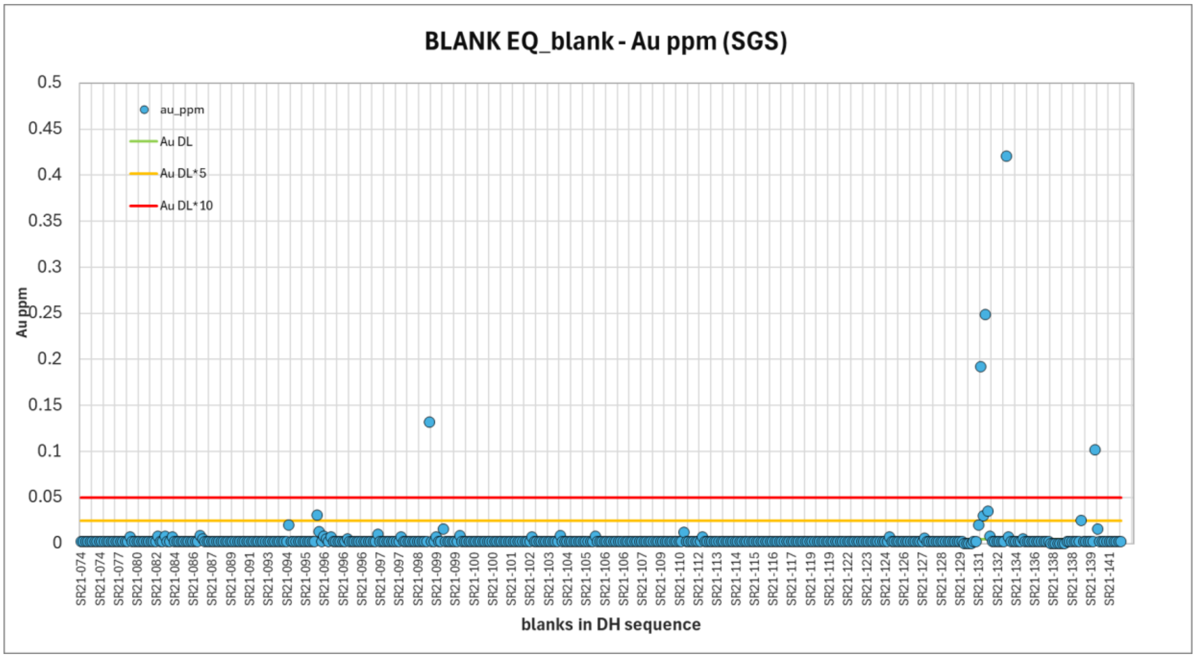


Figure 11-1: Blank 'EQ_Blank' Performance SGS - Au

(Source: MMTS, 2025)

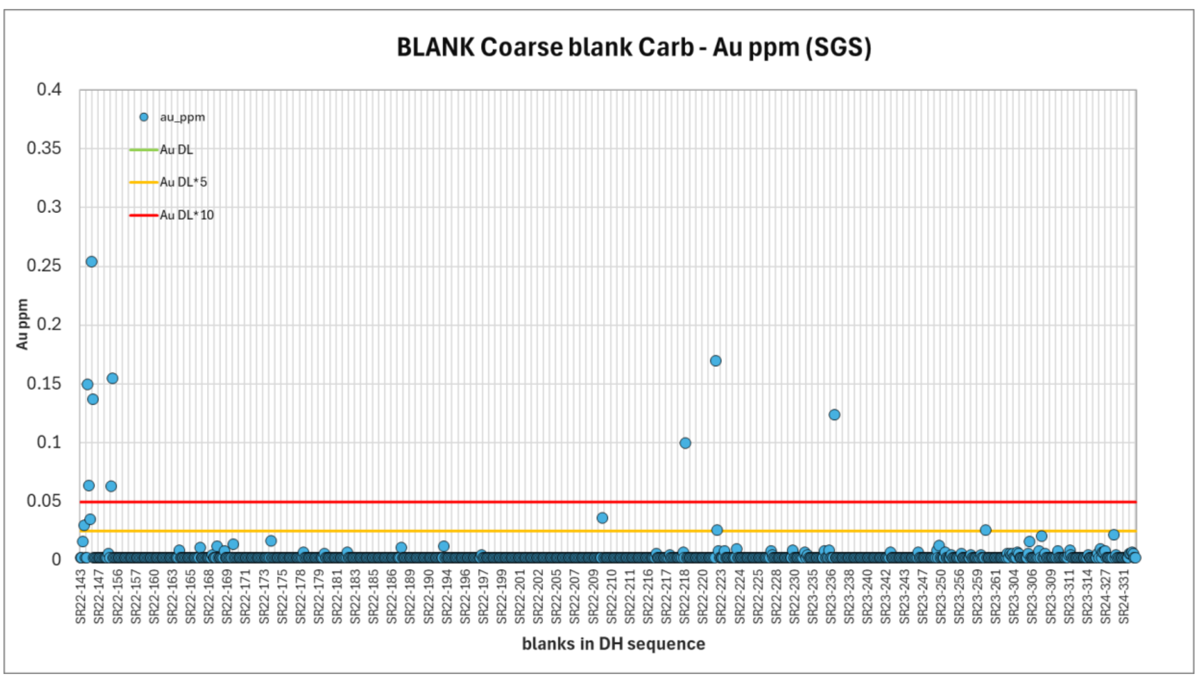


Figure 11-2: Blank 'Coarse Blank Carb' Performance SGS - Au

(Source: MMTS, 2025)

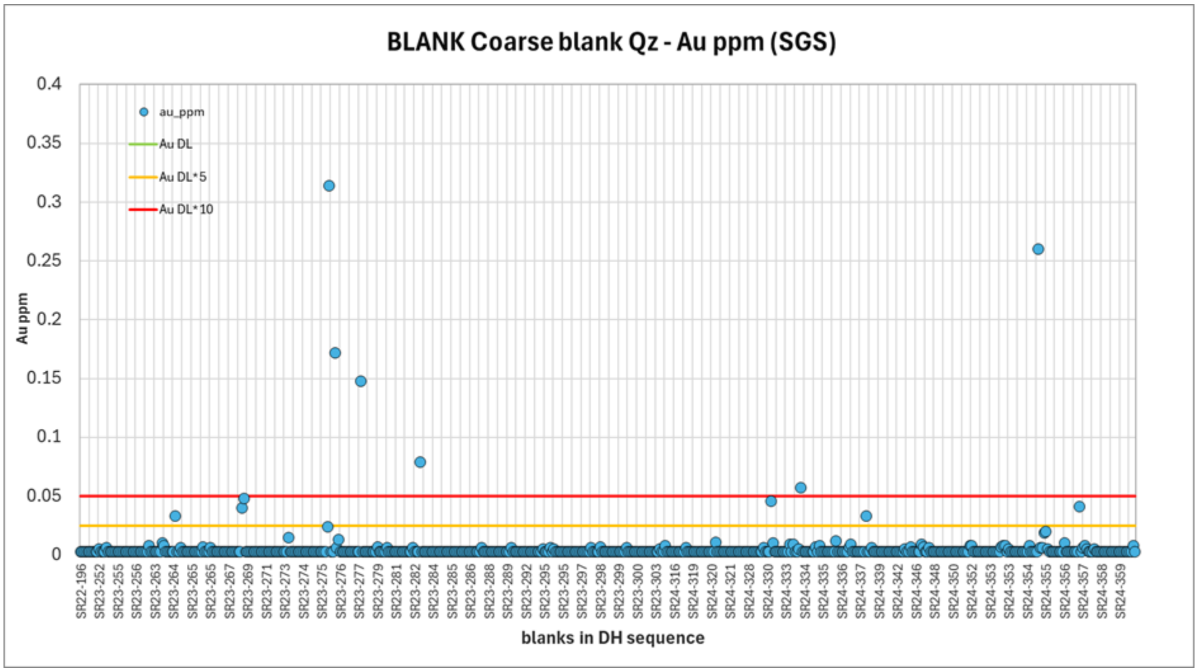


Figure 11-3: Blank 'Coarse Blank Qtz' Performance SGS – Au

(Source: MMTS, 2025)

11.3.5 Standards

This chapter reports on the assay results of blind certified reference materials or standards (CRM or STD) between 2016 and 2024. Four different primary labs were used during this time, and therefore all available STD data for Au and Ag was split by lab to assess and control the accuracy of the data of each lab separately. To combine the results of multiple standards into one graph per metal per lab, the reported assays are normalized into a PCC-style plot with z-score by subtracting the certified (expected) value as per COA (certificate of analysis) from the actual assay result and then divide by the 'in-between' standard deviation (SD), also as per COA. A z-score of +/-2 was defined as warning threshold (orange line) and any +/-3 exceedance is a failure (red line).

In total, eleven CRMs were used from 2019 to 2024. These were purchased from either CDN in Langley, BC, or OREAS in Australia. All used comparable fire assay method data for Au certification. Ten of eleven standards are also certified for Ag using data from both fire assay and 4-acid ICP methods depending on the provider. In 2016, two unknown standards were inserted into the sample stream, see details below.

The CRM details and performances are tabled by CRM and lab which include failure count and rate as well as the percent error on average.

11.3.6 Loring Labs Standards Performance

or 2016, the name, certified u g values, and ‘between-lab’ standard deviations for the two materials that were inserted during drilling and sampling of the SG16 group of holes are currently unknown to MMTS which makes the assay results unsuitable for accuracy control. They were generically named STD-1 and STD-2 for this report and treated as unclassified reference materials (RM) only to demonstrate the overall precision of analyses at Loring Labs in 2016. Means and standard deviations are calculated using the available data (21 Au results for STD-1 and 11 for STD-2) before being normalized. Figure 11-4 and Figure 11-5 graph the acceptably precise Au and Ag results sorted by Drillhole sequence. Table 11-3 provides basic statistics.

Table 11-3: 2016 Loring Labs RM Summary

Standard	Year	Count Au	Au Fails Low	Au Fails High	Au Fails %
STD-1	2016	21	0	0	0.0%
STD-2	2016	12	0	0	0.0%
Total		**33**	**0**	**0**	**0.0%**

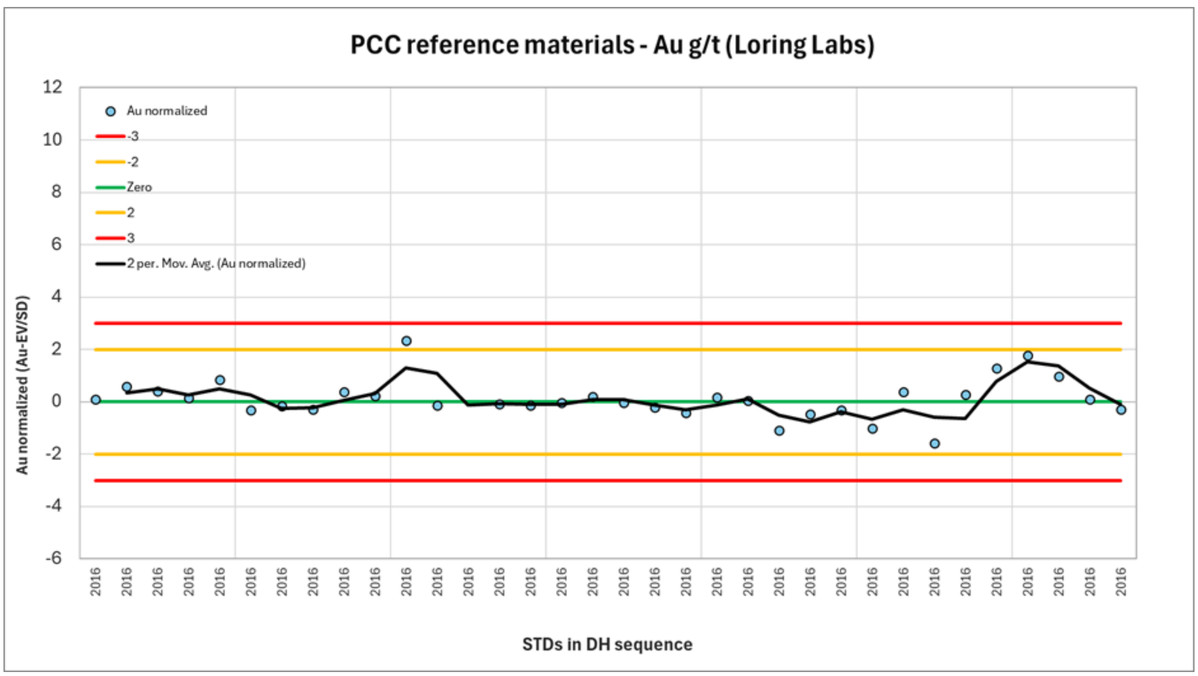


Figure 11-4: 2016 PCC RM Performance Loring Labs – Au

(Source: MMTS, 2025)

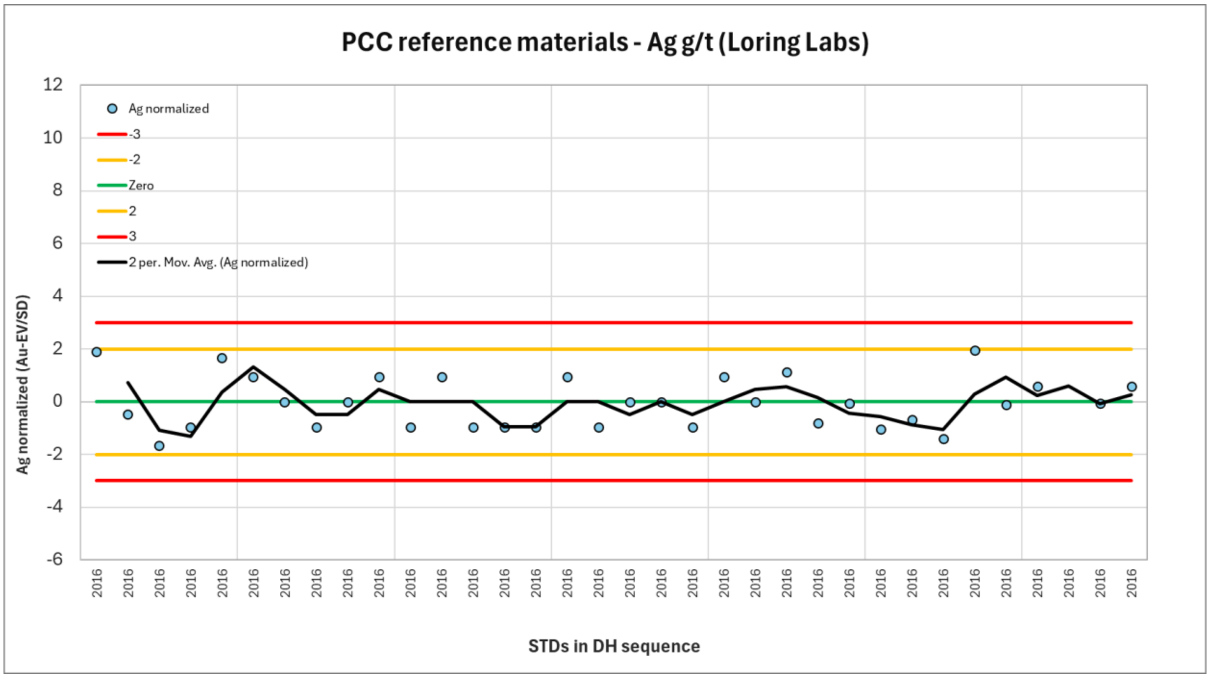


Figure 11-5: 2016 PCC RM Performance Loring Labs – Ag

(Source: MMTS, 2025)

11.3.7 ALS Standards Performance

101 standards were used in 2019-2021 to accuracy-control 1,744 core sample assay results by ALS for an insertion rate of approx. 5.8%. Table 11-4 lists the counts by standard and their respective performances for Au (Ag not shown). The grades range from 0.69 g/t to 27.7g/t Au which is appropriate for the project. The failure rate is 1.0% which is very good. Figure 11-6 and Figure 11-7 visualize the data over time sorted by Drillhole number.

Table 11-4: 2019 – 2021 ALS CRM Summary

Standard	Year	Count Au	Au g/t mean	Au g/t EV	Au g/t SD	Au % Error	Au Fails Low	Au Fails High	Au Fails %
CDN-GS-1P5T	2021	22	1.75	1.75	0.085	0.1%	0	1	4.5%
CDN-GS-25A	2021	5	27.86	27.70	0.450	0.6%	0	0	0.0%
CDN-ME-1501	2020	1	1.33	1.38	0.055	-4.2%	0	0	0.0%
OREAS 228b	2019	23	8.60	8.57	0.199	0.4%	0	0	0.0%
OREAS 523	2019	26	1.04	1.04	0.027	0.2%	0	0	0.0%
OREAS 607	2021	24	0.68	0.69	0.024	-0.9%	0	0	0.0%
Total		**101**					**0**	**1**	**1.0%**

The scatter of the data in Figure 11-6, while within acceptable limits, is noticeably stronger in the 2021 data versus 2019. The calculated 10-sample moving average demonstrates consistent proximity to the expected value (0) and no significant bias.

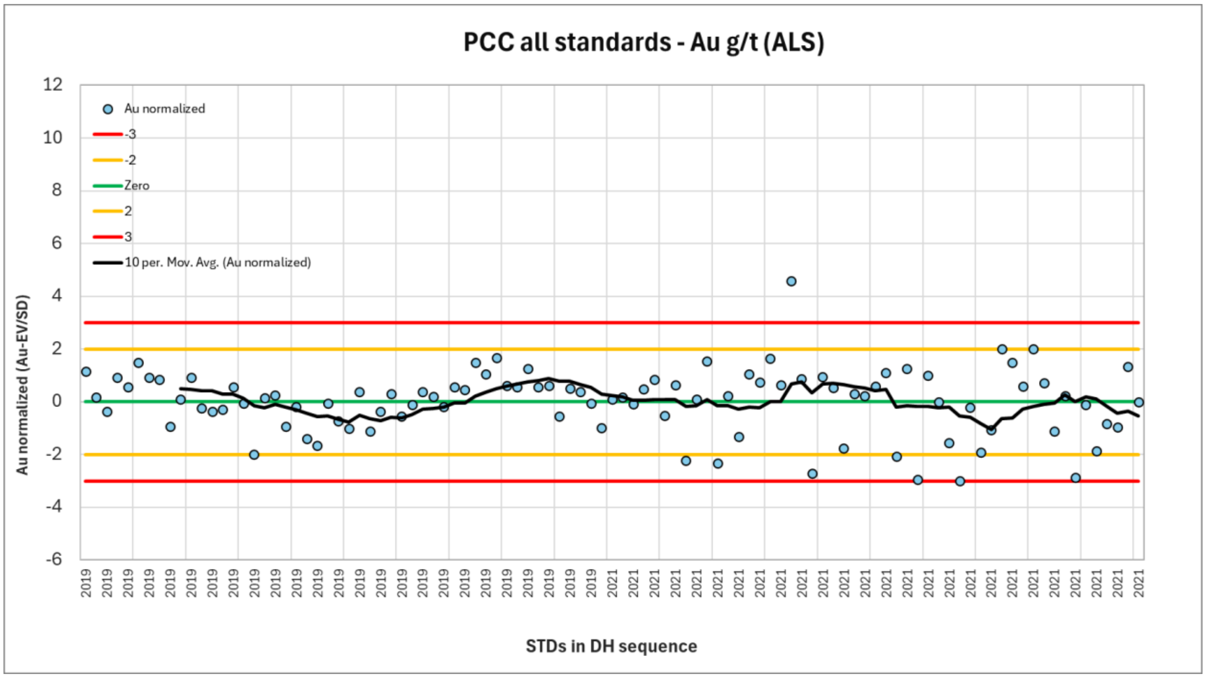


Figure 11-6: 2019-2021 PCC CRM Performance ALS – Au

(Source: MMTS, 2025)

The 9 high failures shown in Figure 11-7 are all the result of OREAS 228b getting reported at 1.4g/t Ag versus the expected value of 1.17g/t. Otherwise, the data demonstrates a moderate high bias in both the ME-ICP41 (2019) and ME-ICP61 (2021) data, but still within acceptable boundaries.

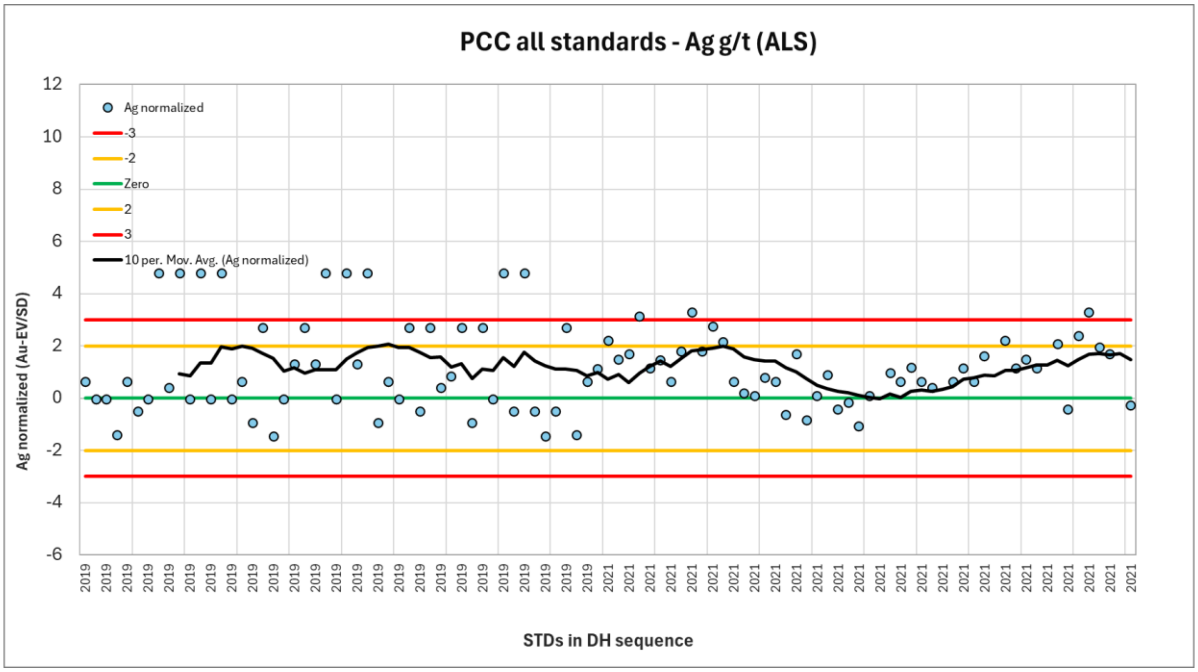


Figure 11-7: 2019 -02021 PCC CRM Performance ALS – Ag

11.3.8 MSA Standards Performance

In 2020, MSA was the primary lab. Table 11-5 summarizes the results for Au. The database contains zero failures on a total of 164 Au analyses which represent the control of 3,594 core samples (insertion rate of approx. 4.6%). 3 different standards were used, one of them an ultra-high Au grade CRM at 25.6g/t that was inserted only once throughout the full drilling campaign despite taking 25 core samples from 15 drillholes that returned >10g/t Au in 2020. The other two (OREAS 523 and CDN-GS-1P5T) are quite similar in grade and as a result do not comprehensively represent the expected Au grade range at the Scottie project. MMTS recommends the usage of at least 3 standards of varying Au grade (low, medium, and high), with the high standard controlling the 'ore-grade' or 'over-limit' method of the lab in a meaningful frequency.

Table 11-5: 2020 MSA CRM Summary

Standard	Year	Count Au	Au g/t mean	Au g/t EV	Au g/t SD	Au % Error	Au Fails Low	Au Fails High	Au Fails %
CDN-GS-1P5T	2020	119	1.74	1.75	0.085	-0.4%	0	0	0.0%
CDN-GS-25	2020	1	25.80	25.60	0.470	0.8%	0	0	0.0%
OREAS 523	2020	44	1.04	1.04	0.027	-0.1%	0	0	0.0%
Total		**164**					**0**	**0**	**0.0%**

Figure 11-8 illustrates very consistent accuracy of Au analyses throughout the 2020 year. No failures, bias, or trends are present. For Ag, the data in Figure 11-9 shows an acceptable performance despite the weak low bias of approx. 1.1%.

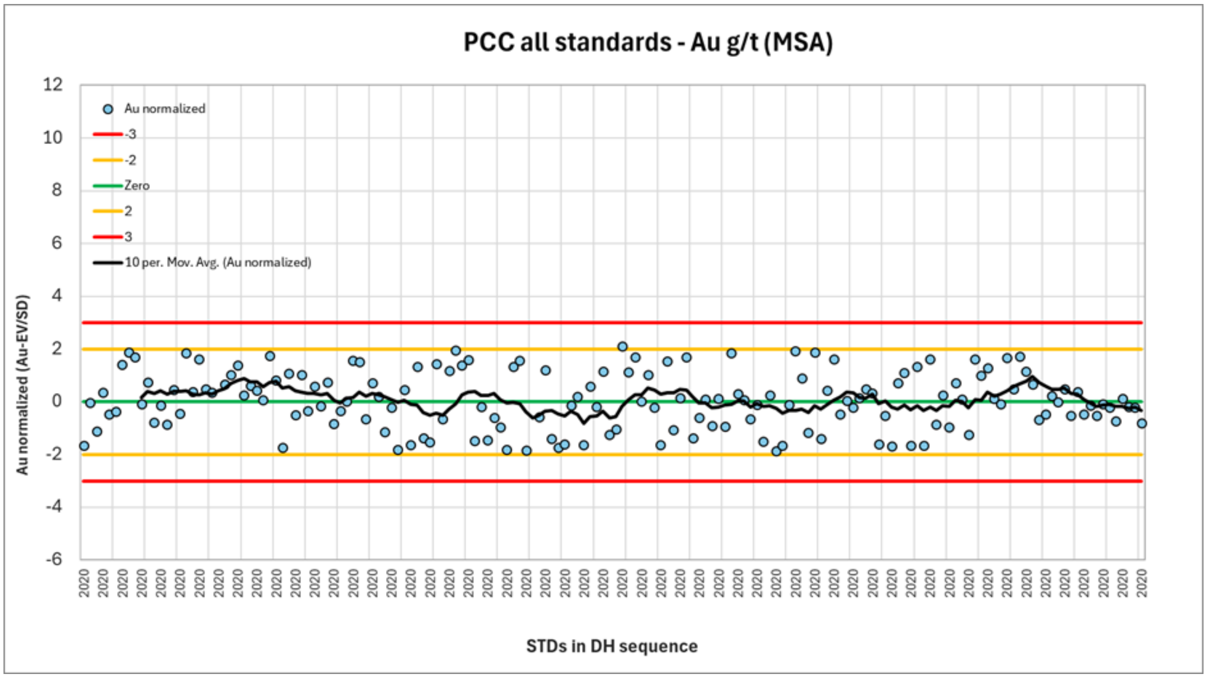


Figure 11-8: 2020 PCC CRM Performance MSA – Au
(Source: MMTS, 2025)

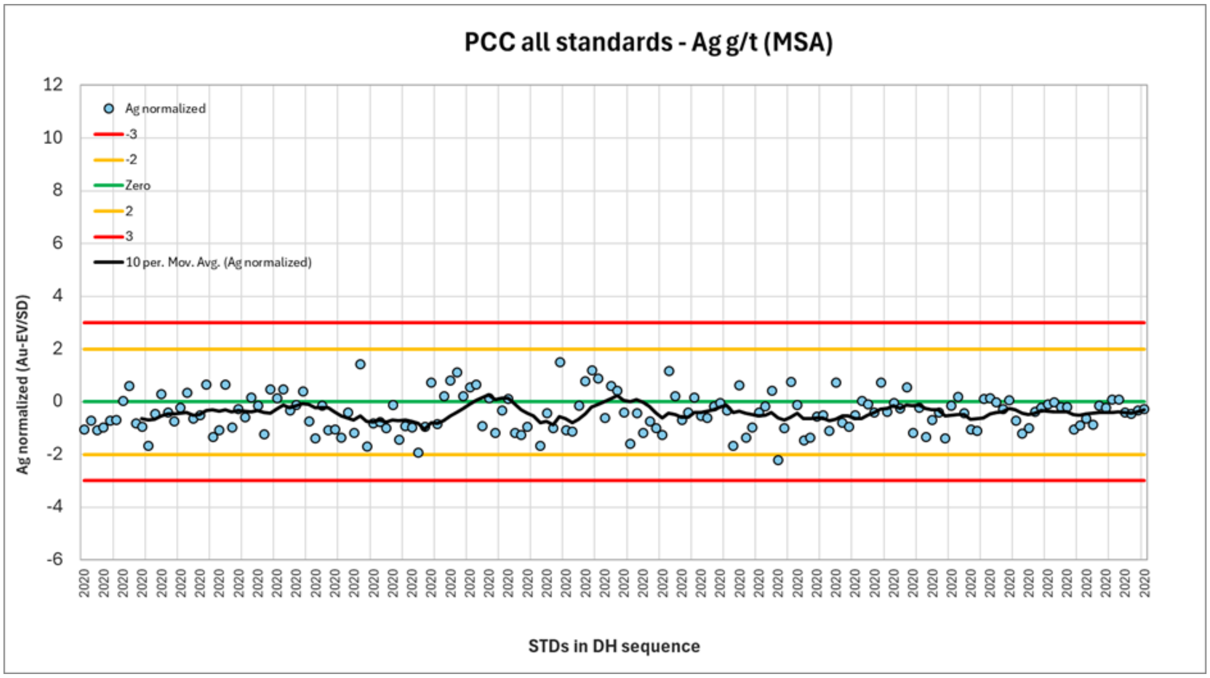


Figure 11-9: 2020 PCCCRM Performance MSA – Ag
(Source: MMTS, 2025)

11.3.9 SGS Standards Performance

By far the most geochemical data between 2016 and 2024 was produced by SGS as it was the primary lab from 2021 to 2024. Almost 35,000 core samples were taken during that time. That total as well as the numbers in Table 11-6 and Figure 11-10 and Figure 11-11 also include the analysis of a few 2020 core samples which were taken retrospectively as gap-filling in 2022.

1,978 total SGS analyses of 9 different 'blind' standards are present in the database to control the u results for accuracy which is an insertion rate of >5%. They represent a grade range from 0.7g/t to 27.7g/t, though strong variations in the grade representation are noted from year to year. At least three different standards were inserted in every drilling campaign.

Table 11-6 lists the details for each CRM with the calculated error between actual (mean) and expected Au value <2.5% for most which indicates acceptable accuracy. Several standards do have a failure rate more than 5% combined, somewhat equally distributed between high and low failures. CDN-GS-25 failed both times it was inserted in early 2021 and was discontinued afterwards, replaced by CDN-GS-25A who performs better at essentially the same ultra-high grade.

The failure rate of the standards population is 3.9%.

Table 11-6: 2020 – 2024 SGS CRM Summary

Standard	Year	Count Au	Au g/t mean	Au g/t EV	Au g/t SD	Au % Error	Au Fails Low	Au Fails High	Au Fails %
CDN-GS-1P5T	2021-2022	226	1.71	1.75	0.085	-2.2%	0	4	1.8%
CDN-GS-25	2021	2	27.80	25.60	0.470	7.9%	0	2	100.0%
CDN-GS-25A	2021-2022	14	27.24	27.70	0.450	-1.7%	3	0	21.4%
sCDN-ME-1501	2022	107	1.41	1.38	0.055	1.8%	1	3	3.7%
CDN-ME-1705	2022-2023	373	3.64	3.62	0.095	0.6%	13	16	7.8%
CDN-ME-1708	2022-2024	35	6.97	6.96	0.250	0.2%	0	1	2.9%
CDN-ME-1903	2023-2024	276	3.02	3.04	0.121	-0.4%	17	10	9.8%
OREAS 607	2020-2023	521	0.69	0.69	0.024	0.5%	2	1	0.6%
OREAS 607b	2023-2024	424	0.70	0.70	0.025	0.3%	3	1	0.9%
Total		**1,978**					**39**	**38**	**3.9%**

Figure 11-10 plots all 1,978 Au results of all standards as analyzed by SGS from 2021 to 2024 as normalized. The failures appear mostly spread out over time except for 2 weeks in October 2022 when several standards across >10 assay certificates reported an increased frequency of variability in Au. The 50-sample moving average (black) mirrors the ideal zero z-score (green) quite well, except for 2022 where the normalized Au data presents a weak high bias of approx. 0.9%. MMTS sees the accuracy control procedures, insertion rates, and results for Au as acceptable.

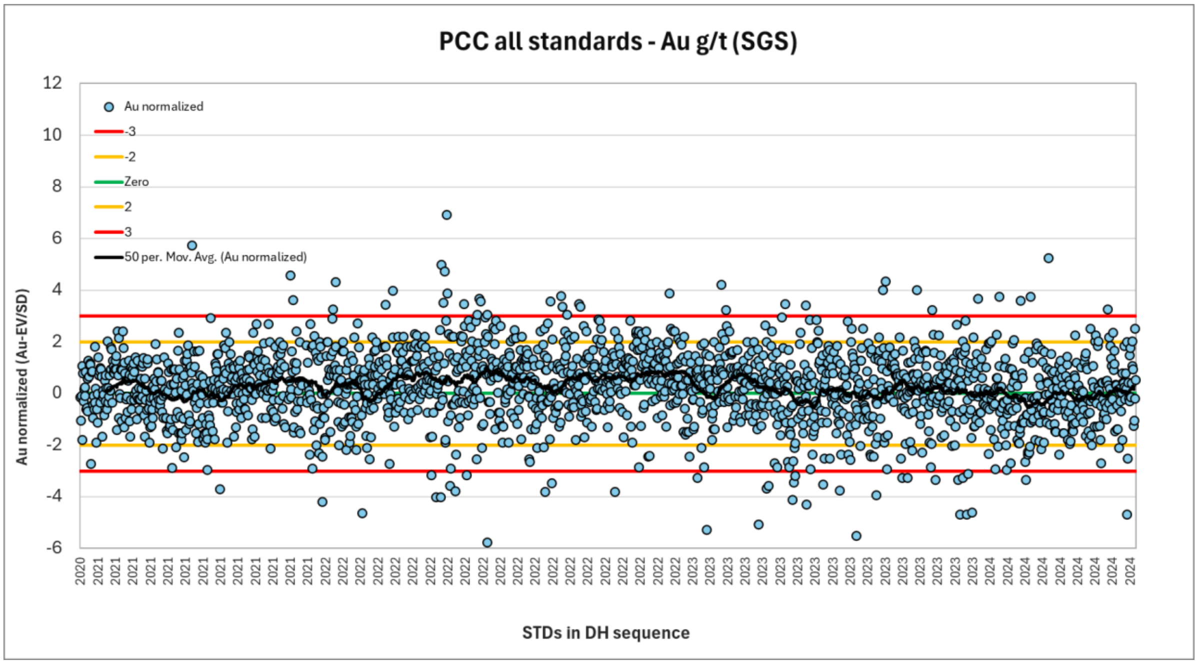


Figure 11-10: 2020 – 2024 PCC CRM Performance SGS – AU

(Source: MMTS, 2025)

The Ag data, also displayed as normalized data in Figure 11-11, shows a segmentation into three parts:

2021 data is biased high with 89 of the 201 inserted CDN-GS-1P5T's plotting above the + -3 failure line at 3.14, and OREAS 607 at 0.63 153 out of 228 times. This is function of the coarse resolution for Ag for data generated by GE_ICP40Q12 (integers) compared to the lower-detection limit GE_IMS40Q12 method that reports down to two decimals.

For 2022 and into 2023, the data is overall unbiased as highlighted by the 50-sample moving average. OREAS 607 again stands out with a consistent z-score of 0.63 (reported actual Ag value of 6 g/t vs. expected value of 5.88 g/t).

The rest of 2023 and all of 2024 data plots slightly biased low because of OREAS 607b who was normalized to -0.44 407 times (96% of all insertions of that CRM), again because of the lack of analysis resolution as mentioned above. Overall, MMTS has no concerns about the accuracy of the Ag data.

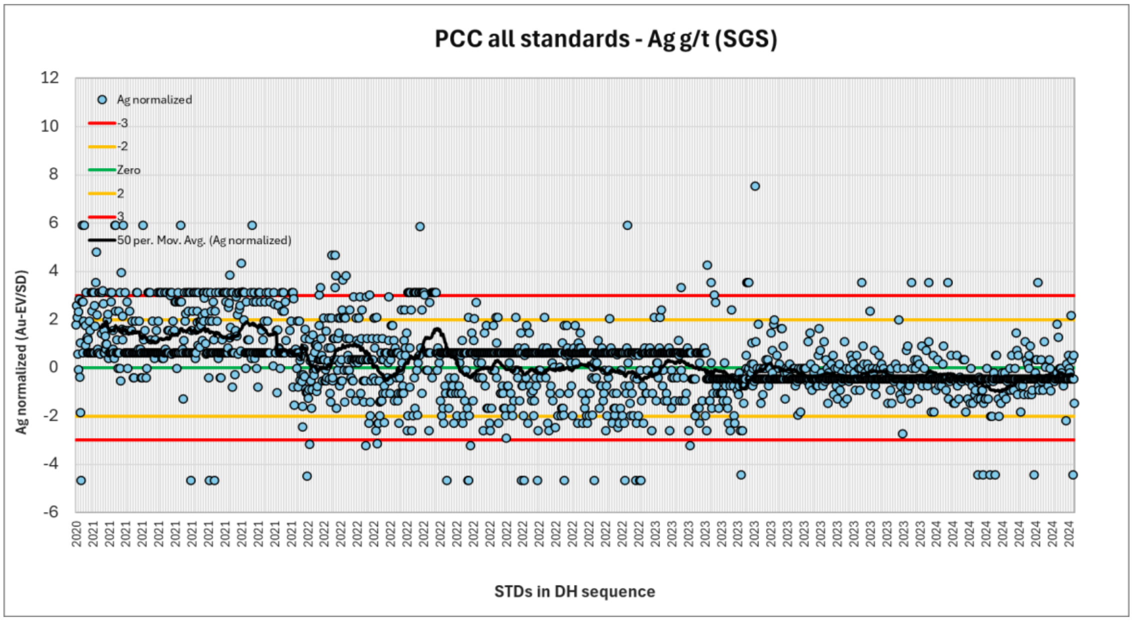


Figure 11-11: 2020 – 2024 PCC CRM Performance SGS – Ag

(Source: MMTS, 2025)

11.3.10 Duplicates

Field, coarse, and pulp duplicates as the means of independent reproducibility or precision control of the various labs contracted for geochemical analyses as mentioned above were introduced in varying frequencies over the last 20 years of the project.

Currently available records of these client-induced duplicates at the Scottie project start in 2004-2005 when both coarse reject and pulp duplicates were requested at relevant frequencies of 3.6% and 16.7%, respectively, though no field duplicates were collected at the time. MMTS has no records of any duplicates taken in 2006.

Loring Labs in 2016 was asked to take coarse and pulp duplicates at rates of 9.9% and 6.1%, respectively, in addition to the lab-internal duplication protocols. Field duplicates were not cut.

The 2019 ALS sample-size reduction process completed in Terrace, BC, was controlled by taking 25 coarse duplicates on 866 total core samples (approx. 2.9%). In 2021, when ALS was the primary lab initially, the coarse duplicate rate was 1.2% (10 on 815 core samples) and 12 field duplicates were also taken (1.5% of all samples analyzed). No pulp duplicates were done in 2019 or 2021.

For the 2020 field season, MSA was the primary laboratory, and Scottie Gold requested both coarse and pulp duplicates be generated at regular intervals and frequencies of about 1 in 45 samples each. 86 field duplicates were also cut from 41 out of 46 drillholes for an insertion rate of 2.3%. The SGS assay data from 2020-2024 (includes gap sampling of 2020 core in 2022) contains 732 field duplicates distributed over all 5 years of sampling and several of the known prospects at Scottie for an insertion rate of 2.1%; coarse duplicates for 2021 sum to 104 total (approx. 1.4% of all drill core samples). No pulp duplicate records are available.

MMTS has not reviewed lab-internal duplicates.

11.3.11 2020 – 2024 Field Duplicates

Field duplicate sample assays just like field original and primary sample assays contain all potential size- reduction errors introduced by the lab during preparation of each sample as well as any analytical error. Most importantly though they all contain the inherent mineral and therefore chemical variability of the tested mineralized system at hand, a variability that is generally influenced by geological factors like mineral grain size, textural complexity, possibly paragenesis, and certainly width or thickness of the mineralization. Core sampling is generally based on the geological descriptions, e.g. determination of lithological contacts among other records, which can limit flexibility in choosing the ideal sample size (core length) from a reproducibility perspective. Another limiting factor is the drill bit diameter, the choice of which is often dependent on external framework like cost, terrain, equipment availability, etc.

For this report, MMTS combined limited field duplicate assay data as reported by ALS (12 results in 2021) and MSA (84 in 2020) with 729 data points produced by SGS between 2021 and 2024 (Figure 11-12).

Hydrothermal precious metal systems like the Scottie project are often constrained to relatively small veins, faults, or breccias, and high-grade Au or Ag intersects tend to be a function of drilling and sampling comparatively few coarse, yet often not visible, and unevenly distributed precious metal grains within these mineralized structures. This can lead to sample reproducibility concerns, which increase with small drill core diameter and decreasing sample interval length.

At Scottie, field duplicates are predominantly being produced by quarter-cutting the NQ-sized core of the selected interval so that both original and duplicate are being represented by one quarter each. Ideally, this results in very comparable sample weights. The average field duplicate core length for the years 2021-2024 is 1.66 m with sample weights for original core and duplicate core averaging 1.80 kg and 1.82 kg, respectively, which is very good. For 2020, the field originals are being represented by half core (sample weight average of 2.49 kg) while the duplicates are quarter core (avg. of 1.31 kg). The average interval length is also significantly shorter at 1.24 m.

Coarse duplicates are taken after the crushing stage when an additional approx. 200 g sample is generated during splitting, while pulp duplicate assays represent two cuts from one already pulverized sample with individual digestions and analyses. The further along the sample size reduction process the duplicate is taken, the higher the correlation to the original sample is expected to be.

The correlation of 825 field original-duplicate pairs is shown in a simple logarithmic scatter plot in Figure 11-12 for Au. While the R^2 of 0.75 for Au proves acceptable reproducibility for >95% of the data, a small population of 15-20 duplicates does demonstrate significant scatter at meaningful grades >0.1 g/t Au. Several of those ‘outliers’ were checked for sample, misclassification, or data entry errors, but it appears that they likely represent un-biased natural variability of Au in mineralized rocks at the Scottie project.

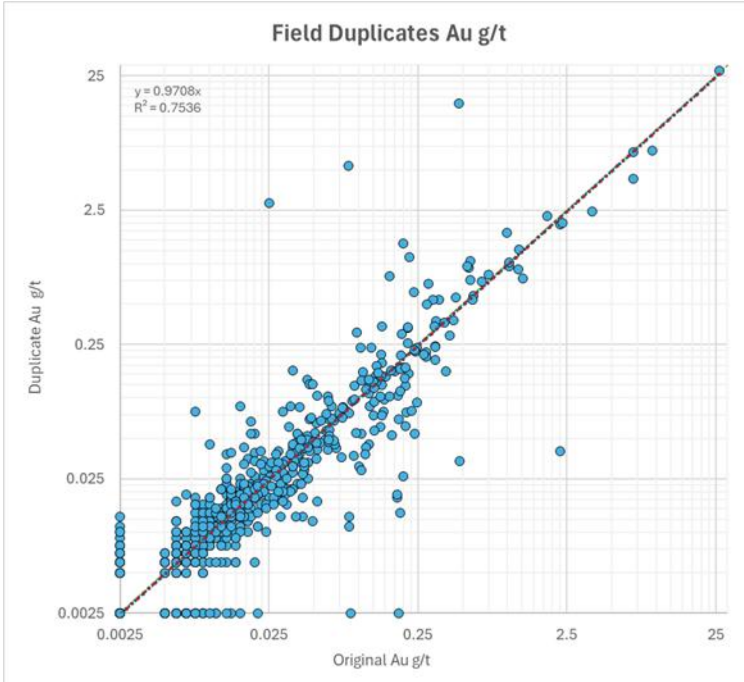


Figure 11-12: 2020 – 2024 Field Duplicates Scatter Plot – Au
(Source: MMTS, 2025)

11.3.12 Coarse Duplicates

Coarse duplicate performance is a direct indicator of how suitably the preparation department of the respective labs crushed, homogenized, and split the received core samples, and the data certainly contains pulverizing and analytical errors, if any. Crushing to an industry-standard of 70% passing 2 mm generally allows the lab to representatively split the sample into sub-sets with acceptable precision even in a high-grade Au system like Scottie.

The following plots in Figure 11-13 to Figure 11-16 demonstrate reproducibility of Au results at this sample size-reduction stage for each of the 4 labs contracted to do the prep work between 2016 and 2024. The data presents analyses of coarse duplicate material selected by the operating company, presumably to assure Au grade representation in addition to the standardized lab-internal coarse duplicate insertions which are sufficiently frequent but without relation to the quality or grade of the sampled material.

MMTS did not review the lab-internal QA/QC duplicates.

The grade range in coarse duplicates of Loring Labs 2016 tops out at 5 g/t (Figure 11-13) and the overall sample distribution is skewed towards low-grade samples <0.25 g/t Au but the correlation of the dataset is acceptable.

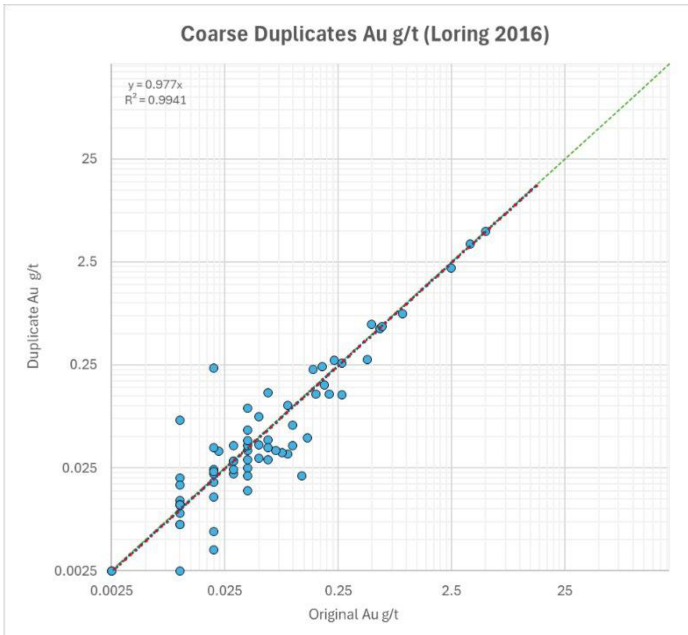


Figure 11-13: 2016 Coarse Duplicates Scatter Plot Loring Labs – Au
(Source: MMTS, 2025)

The 35 Au data points in Figure 11-14 show coarse duplicate performance at ALS in 2019 and 2021. The grades range from <DL to 8.5g/t and the correlation coefficient is very good (R^2=0.98).

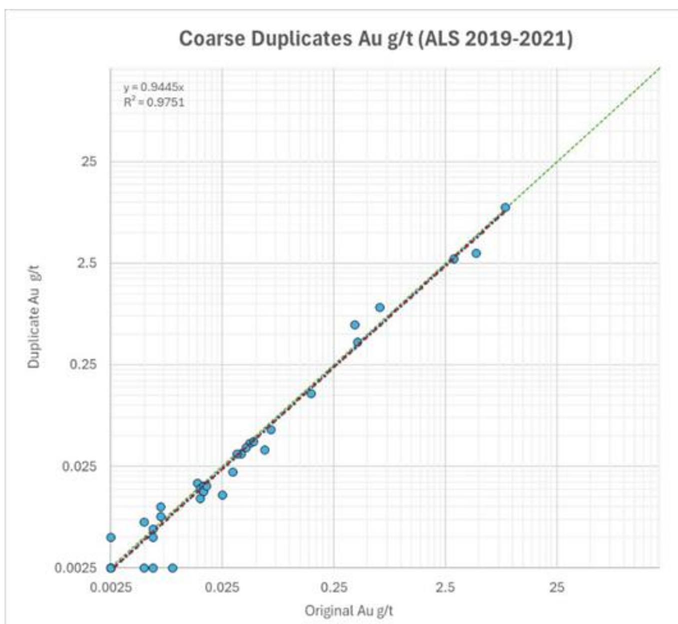


Figure 11-14: 2019 – 2021 Coarse Duplicates Scatter Plot ALS – Au
(Source: MMTS, 2025)

In 2020, 84 coarse duplicates were analyzed for Au at MSA (Figure 11-15). The grades range from <DL to a very high-grade >200g/t. The correlation coefficient is perfect (R^2=1), influenced by the one very high Au pair taken in Drillhole SR20-048 which returned 205.4 and 204.6g/t Au for the original and duplicate, respectively.

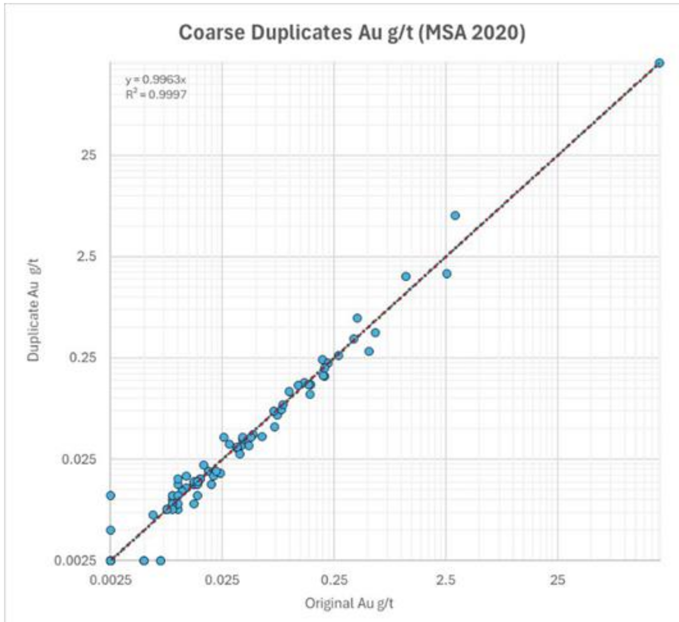


Figure 11-15: 2020 Coarse Duplicates Scatter Plot MSA – Au

(Source: MMTS, 2025)

In 2021, SGS produced 101 coarse duplicate analyses for Au (Figure 11-16). The correlation is very good (R^2=1), indicating that reproducibility is acceptable at selected sample size reduction procedures at SGS, and the grade range is representative of the Scottie gold system. Scottie Gold has not requested coarse duplicate splits during the 2022-2024 drilling campaigns.

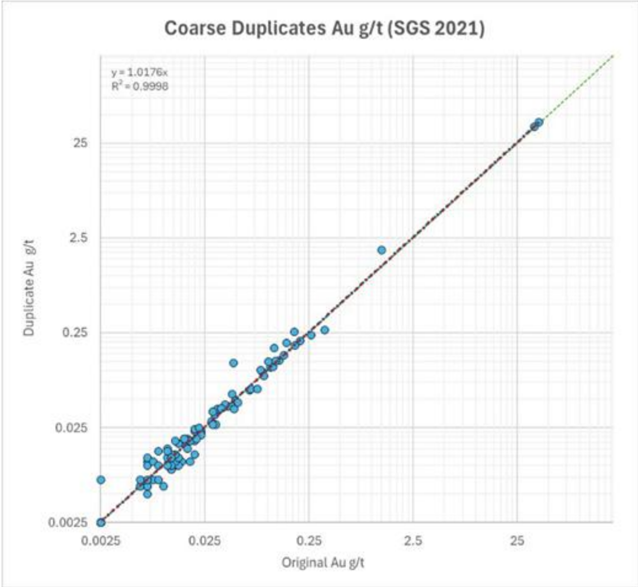


Figure 11-16: 2021 Coarse Duplicates Scatter Plot SGS – Au

(Source: MMTS, 2025)

11.3.13 Pulp Duplicates

Pulp duplicates requested from Loring Labs in 2016 performed acceptably (Figure 11-17), especially in the relevant Au range of 0.25-25 g/t. No significant outliers are noted, and the correlation is very good as indicated by R^2 at 0.99. The noticeable scatter in the data <0.1 g/t indicates analytical precision challenges at low Au concentrations at Loring.

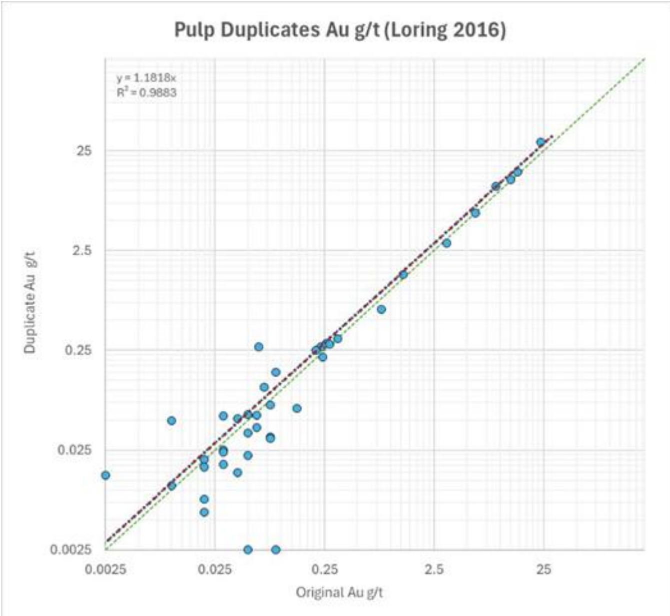


Figure 11-17: 2016 Pulp Duplicates Scatter Plot Loring Labs – Au

(Source: MMTS, 2025)

In 2020, 82 pulp duplicates were requested from MSA (shown in Figure 11-18). The Au results correlate well with the Au grades in the corresponding pulp original (R^2=1). The grade representation is also very good from DL to >25 g/t.

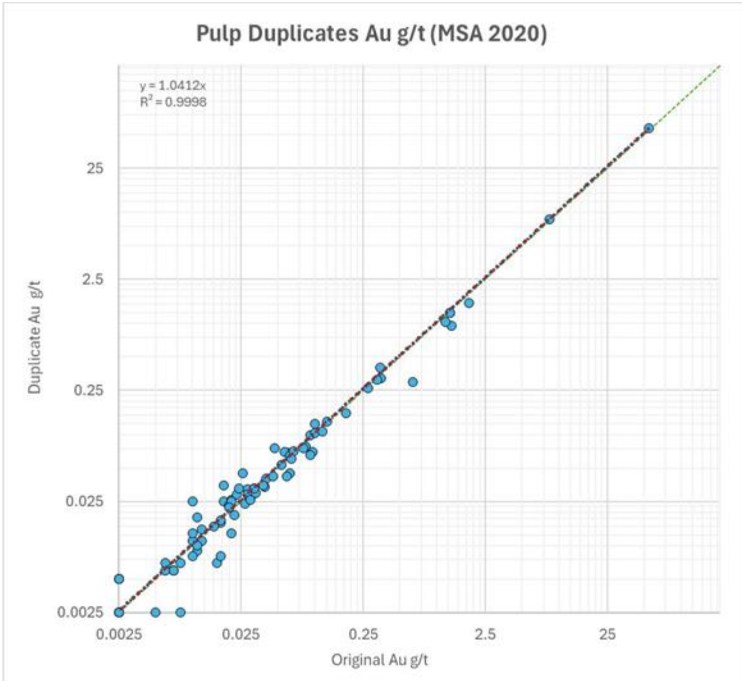


Figure 11-18: 2020 Pulp Duplicates Scatter Plot MSA – Au

(Source: MMTS, 2025)

Scottie Gold did not order pulp duplicates from ALS (2019 and 2021) or SGS (2021-2024). MMTS did not review the lab-internal pulp duplicates/replicates for any of the four labs.

11.3.14 Check-Assays

78 pulps produced and analyzed by SGS from core samples taken during the 2021 drill campaign were sent to MSA Labs for Au check assay purposes, with FAS-111 (30 g fire assay with AA finish) and FAS-415 (gravimetric) requested depending on grade reported by the primary lab. The two resulting datasets correlate very well with an R^2 of 0.98 without significant bias. One single sample (D740546) records as an outlier (6.2 g/t at SGS vs. 18.4 g/t at MSA) and probably represents a sample mix-up.

11.3.15 QA/QC Historical

Only limited blind QA/QC data is currently available for the Scottie project pre-2016. This includes the use of blanks, (unknown) standards, and coarse and pulp duplicates in 2004-2006. MMTS is not aware of field duplicate data or suitable blind QA/QC of any kind prior to 2004. However, a limited number of Au analyses for core (and rock samples) were reported by N.L. Tribe and Associates (1983), that allow a comparison of assay results between the two labs used at the time (Scottie Lab and Premier Lab) as well as a secondary certified lab in North Vancouver (Vangeochem Lab. Ltd) that was used for check-assaying. This data is presented under Section 11.6.4.

EcoTech out of Kamloops, BC, was the primary lab in 2004-2005 and possibly 2006 as well even though no assessment report is available to confirm that year.

In 2018, Scottie Gold had selected coarse rejects of the 2004-2005 drilling campaigns re-processed and re-assayed at ALS in Vancouver, BC, the results of this exercise are shown under Section 11.6.4. This was part of a larger relogging and resampling program that has been described in detail by Ron Voordouw and Ian Carr of Equity Exploration Consultants in the 2018 Geological and Geochemical Report on the Scottie Gold Mine Project (2019). The resampling efforts produced some 52 samples from historically sampled core, however due to previous relogging, the sample intervals did not match the historical ones and as a result, the Au grades could only be indirectly compared, for example through data compositing. Correlations of Au data turned out to be poor (Equity 2019).

Table 11-7 details QA/QC insertions for the 2004-2006 seasons.

Table 11-7: Historical QA/QC Insertions Summary

Year	Lab	Core Samples	Blanks	CRMs	Field Dup	Coarse Dup	Pulp Dup	QAQC Total	% QAQC	Check	Comment
2004	EcoTech	506	20	20	0	13	67	120	19.2%	1	
2005	EcoTech	902	39	39	0	38	169	285	24.0%	9	
2006	UNK*	897	4*	5*	0	0*	0*	9	1.0%	0	*Data incomplete
Total		**2,305**	**63**	**64**	**0**	**51**	**236**	**414**	**11.9%**	**0**	

11.3.16 Blanks

Figure 11-19 graphs all available 2004-2006 Au assay data for 63 inserted blind blanks. No failures are recorded despite at least three of the blanks directly succeeding high-grade Au intervals of grades between 9 and 20 g/t, proving that cross-sample contamination is not a concern for the assay data generated by EcoTech. MMTS could not locate blanks assay results for most of the 2006 drillholes (DH 737-762).

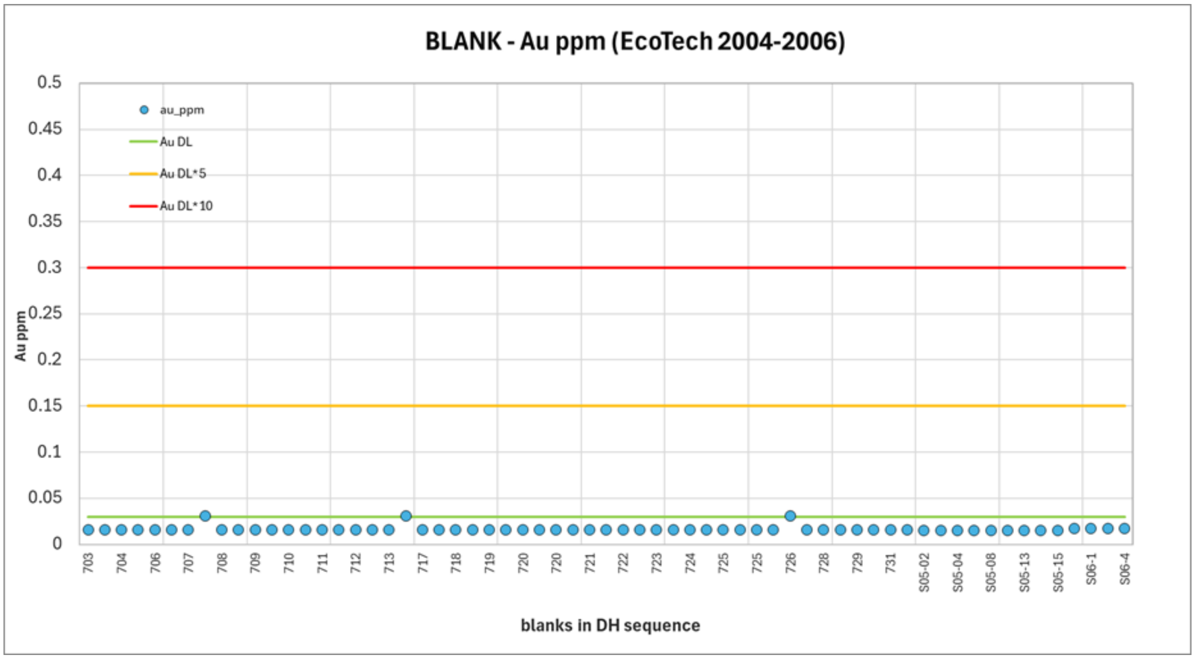


Figure 11-19: 2004 – 2006 Blank Performance EcoTech – Au

(Source: MMTS, 2025)

11.3.17 Standards

64 assay results for 5 different standards are available for the period 2004-2006. Unfortunately, three of the five standards names and certification details are currently not known to MMTS and are therefore being viewed as reference materials (RM) for control analytical precision only. In the interim and for the purpose of this report, they are named STD-3, STD-4, and STD-5.

Figure 11-20 shows all five reference materials over time, with the assay data normalized by using the average grade and standard deviation for each population. Just like with the blanks and duplicates, reports or datafiles about most accuracy control samples for the year 2006 could not be identified.

The data indicates overall acceptably precise results with one theoretical high failure in Drillhole 714 but without the required certification of the material it can not serve as industry-standard accuracy control at this point. Ag data is not available.

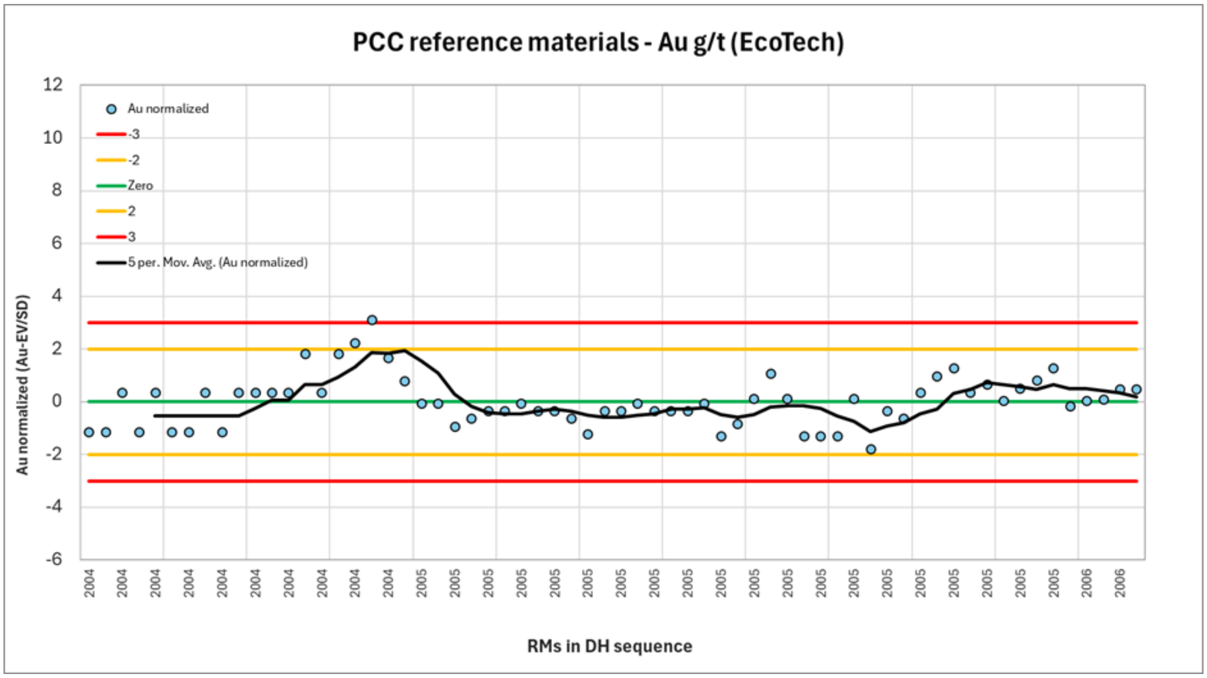


Figure 11-20: 2004 – 2006 PCC CRM Performance EcoTech – Au
(Source: MMTS, 2025)

11.3.18 Duplicates

The Scottie Gold assay database does not contain any field duplicate data pre-2020 but both coarse and pulp duplicates were inserted at appropriate rates for 2004 and 2005 (MMTS has no records of 2006 QA/QC data or copies of 2006 original certificates). According to ARIS report 28190, at least for the 2005 drill campaign, the core samples were handled at the EcoTech prep facilities in Stewart, BC, before the resulting pulps were sent on to EcoTech in Kamloops, BC, for digestion and analysis.

11.3.19 Coarse Duplicates

Figure 11-21 illustrates the very good correlation between coarse original and coarse duplicate results in 2004-2005. Weak scatter at very low grades is not a concern and the grade representation between 0.1 g/t and 25 g/t Au is near perfect for such a small sample size (50 pairs on 1,408 total core samples, 3.6% insertion rate).

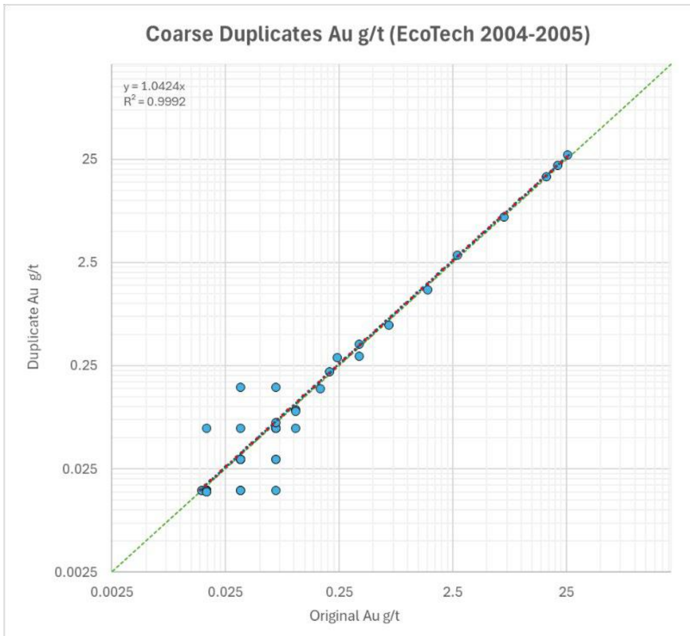


Figure 11-21: 2004 – 2005 Coarse Duplicates Scatter Plot EcoTech – Au
(Source: MMTS, 2025)

11.3.20 Pulp Duplicates

The scatter plot in Figure 11-22 shows the Au assays of 228 pulp duplicates taken in 2004-2005 versus their 228 pulp originals. The correlation is perfect as indicated by R^2=1, proving very high analytical precision, and the grade range as well as the sample distribution within that range is also very good.

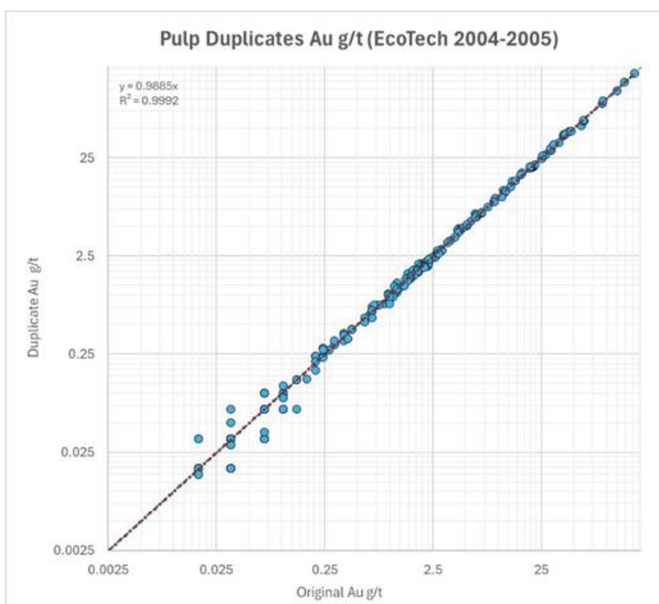


Figure 11-22: 2004 – 2005 Pulp Duplicates Scatter Plot EcoTech – Au
(Source: MMTS, 2025)

11.3.21 Check-Assays

As part of a re-logging and re-sampling campaign in 2018, 10 coarse rejects from the 2004 and 2005 drilling campaigns were selected for modern-day analysis to compare against the fire-assay Au results from EcoTech at the time. 9 of the 10 rejects are high-grade material >5g/t Au (see Figure 11-23). The correlation as illustrated by R^2 (0.97) is very good for a coarse reject re-analysis by a separate lab more than 10 years later, though a weak ALS-positive bias carried by 7 of 10 results is noted. However, MMTS does not consider a population of 10 assay results representative enough to understand if historical EcoTech Au results should be seen as consistently conservative.

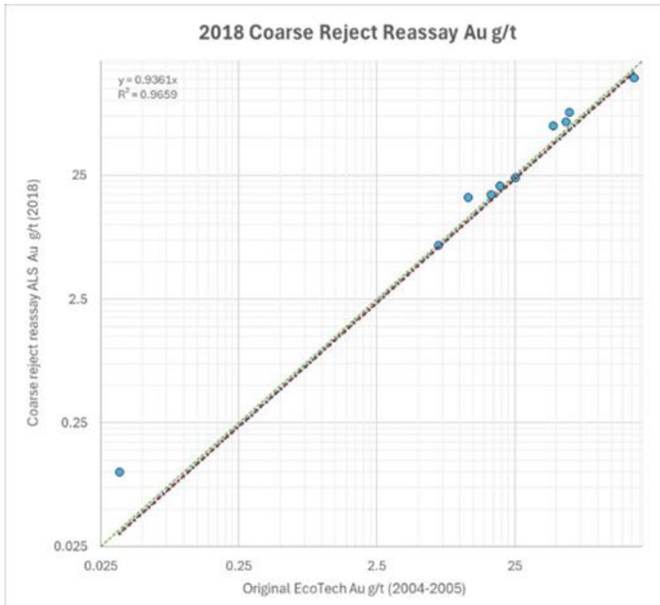


Figure 11-23: 2018 Coarse Reject Re-assay EcoTech vs. ALS – Au

(Source: MMTS, 2025)

The general process of check-assaying as performed in 1983 was described under 11.4.2. The small population of 44 data points in Figure 11-24 indicates that the analyses completed at the Scottie lab in Stewart, BC, were generally higher at lower grades to approx. 0.1 opt while the high to very high grades >0.1 opt turned out to be very comparable between the Scottie lab and the secondary (check) lab at the Premier Mine. The very good correlation shown in the graph (R^2=0.99) is carried by these high-grade pairs.

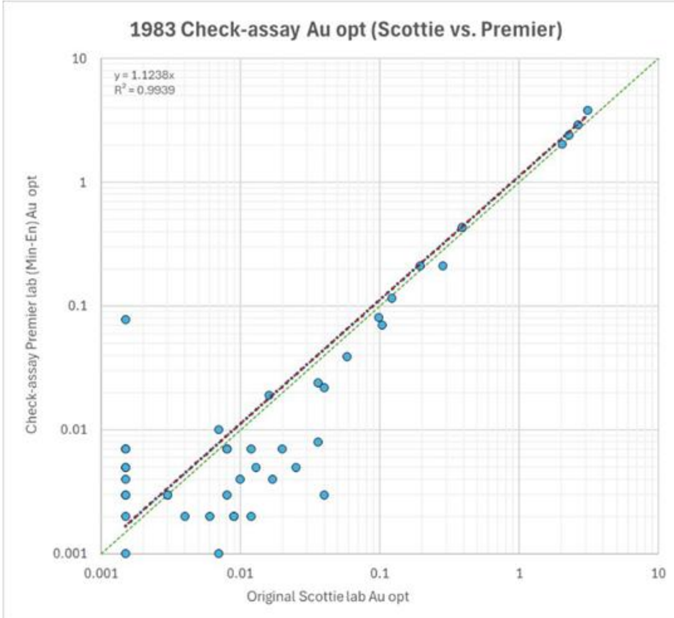


Figure 11-24: 1983 Check-Assay Scottie vs. Premier – Au

(Source: MMTS, 2025)

Figure 11-25 shows a scatter plot of 25 samples that were analyzed for Au at the Scottie lab, then afterwards check-assayed at Vangeochem in North Vancouver, BC. The data displays an overall very good reproducibility, except for 3 samples that clearly qualify as strong outliers. Given that fully prepped pulps were sent to the secondary lab, these poor results may only be explained by sample number or other database errors at the time.

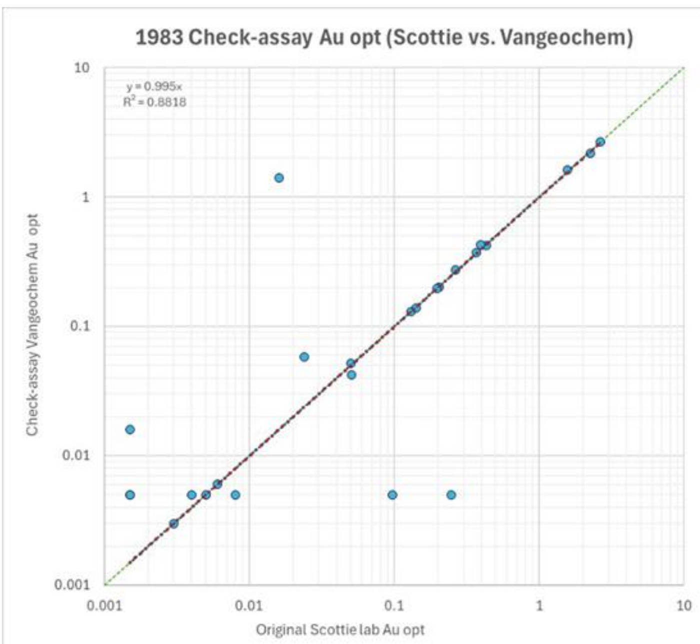


Figure 11-25: 1983 Check-assay Scottie vs. Vangeochem – Au

(Source: MMTS, 2025)

11.4 Summary and Conclusions

Since Scottie Resources became operators of the Scottie project in 2018, geochemical data as available for this report is sufficiently quality assured and quality controlled by the insertion of field duplicates, blind blanks and blind standards from 2020 to 2024.

12.0 DATA VERIFICATION

This section summarizes the verification work and practices employed by Scottie Resources Corp. and previous operators of the Scottie Gold Mine Project. The independent Qualified Person (QP) responsible for Section 12 of this report, Sue Bird, P. Eng., believes the databases are sufficiently validated and verified to support their use in mineral resource estimation for each of the deposit as presented herein.

12.1 Site Visit

A site visit was conducted on September 7, 2025, by Sue Bird, P.Eng. of MMTS for one day. During the site visit the Blueberry, Bend and Scottie Mine deposit sites were all visited. Several drillholes at the Blueberry site were surveyed for location verification and the Blueberry contact examined. The drilling that was occurring at the Scottie Mine site during the site visit was flown over and hole locations verified. Core handling and storage at the camp was reviewed, as was the overall geology of each deposit. It was not possible to go underground, but several adits were noted during the fly-over. The core storage and office site in Stewart was also toured with 7 samples taken for re-assay as checks on previous drill results.

Figure 12-1 illustrates the camp site showing the camp site and the old mill infrastructure. Figure 12-2 shows the core logging area and the core boxes used for transportation to the assay lab, with Figure 12-3 showing the core storage area in Stewart.

Figure 12-1: Overview of Scottie Project Camp

(Source: MMTS, 2025)

Figure 12-2: Logging Area and Drill Core Boxes at Site

(Source: MMTS, 2025)

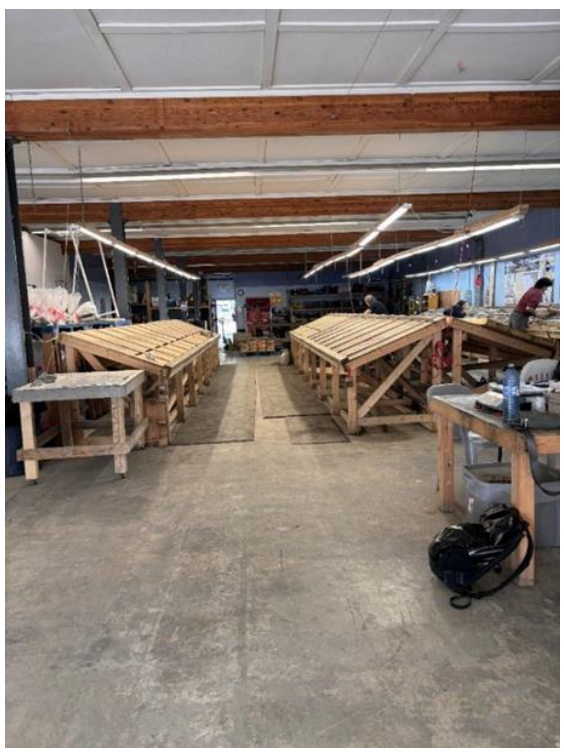


Figure 12-3: Core Logging Warehouse in Stewart

(Source: MMTS 2025)

12.2 Re-assay Results

Seven samples for re-assaying were collected during the September 2025 site visit. The results of this check assaying are provided in the plot below (Figure 12-4), showing generally good correlation with the original assays.

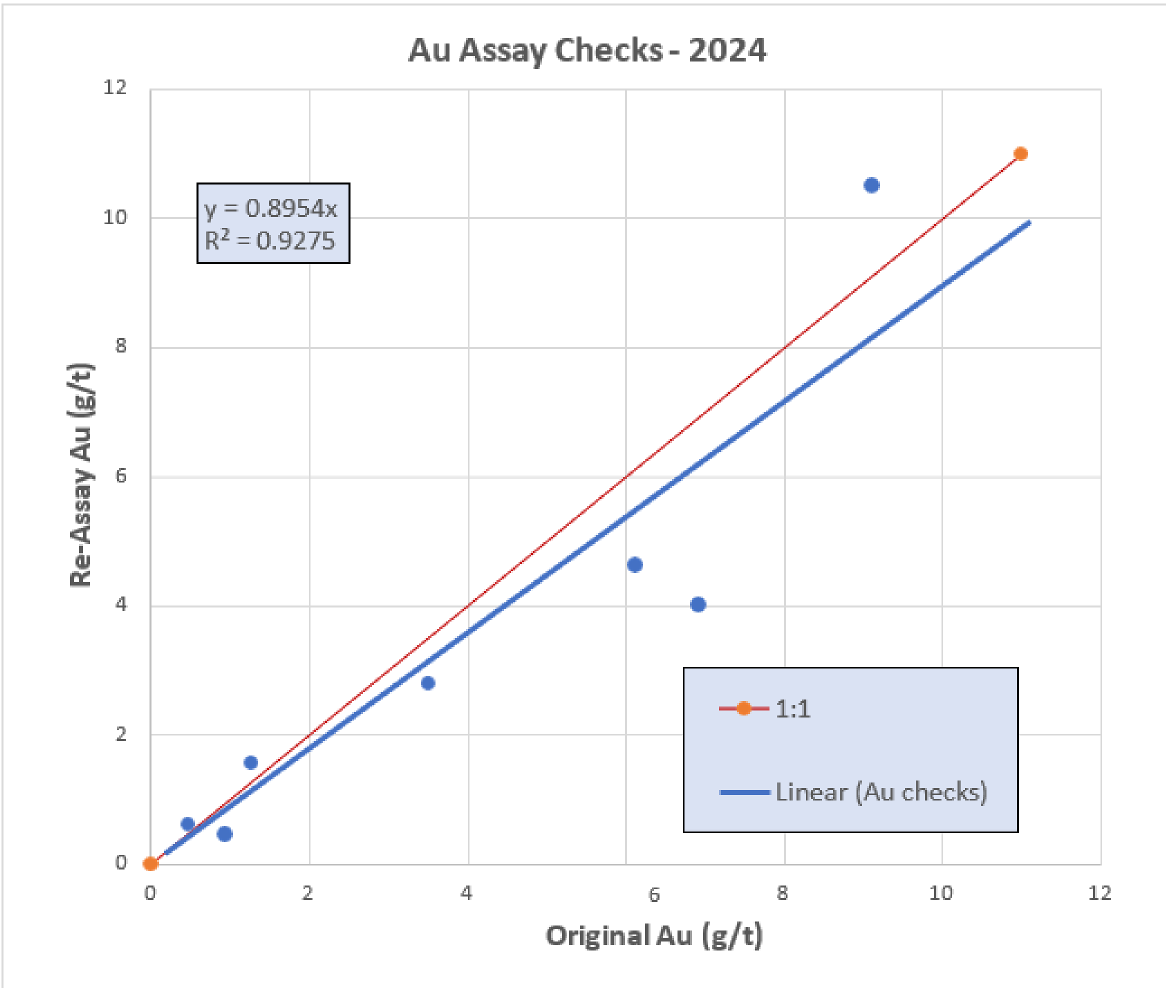


Figure 12-4: Check Assay Results from 2022 and 2024 Site Visits – Au

(Source: MMTS: 2025)

12.3 Data Audit

Certificates checks on the Au grades within the mineralized sone were completed on the historic data from Assessment reports. Only minor discrepancies were noted.

12.4 Validation of Historical Data

A significant portion of the Scottie Mine deposit data is historical and Certificate Data was not available. This data has been validated through “point validation”. This is a statistical method in which the historical data only is used to interpolate the expected grade to the location of the recent Au composites (with Certificates), in order to essentially remove the spatial variability factor and therefore help determine if there is any bias existing in the historical data. Figure 12-5 illustrates this comparison for historical data within the current mineralization shapes and within 50 m of the recent drilling. The historical interpolated data shows more smoothing (as expected) and also illustrates a potential low bias of the higher grades and median grades (grade above approximately 0.6 g/t Au). Data within 10 m was also interpolated and showed a similar result.

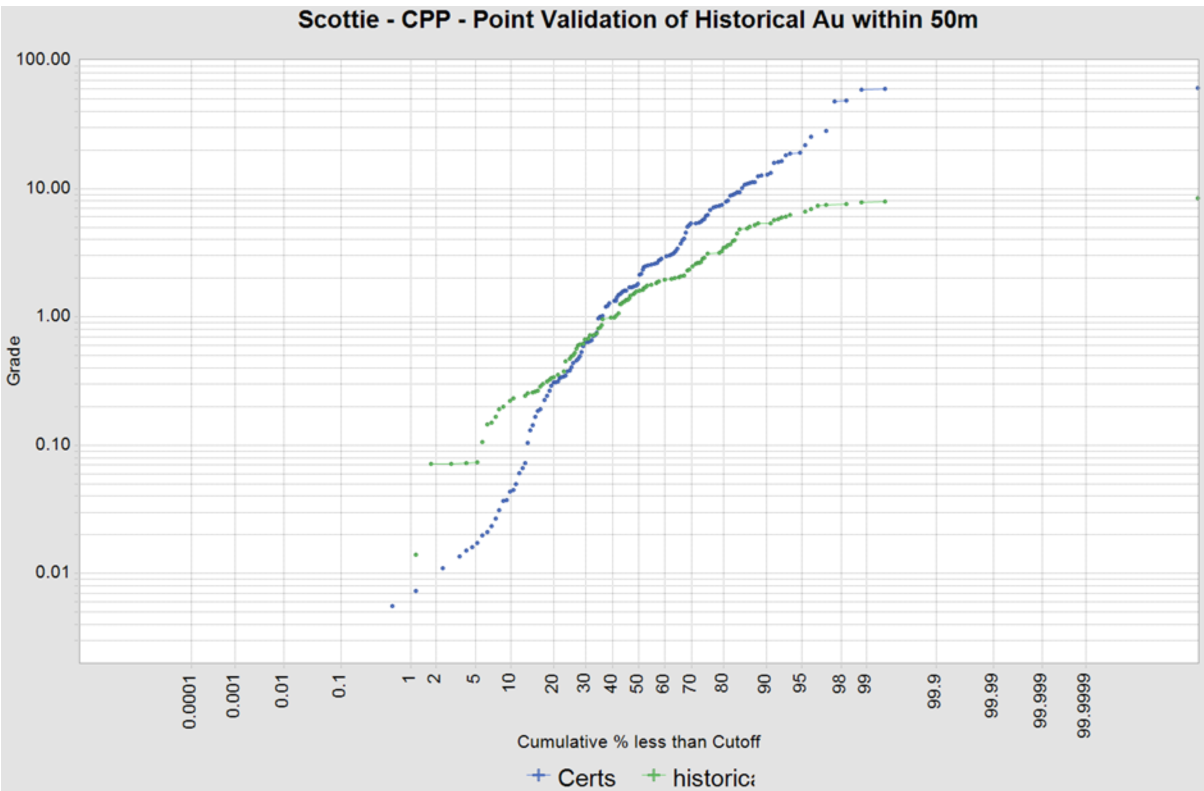


Figure 12-5: Comparison of Au Grade Distribution with Certificates and Historical Data

12.5 Data Verification Conclusions

12.5.1 Resource Estimate

Sue Bird, P.Eng. of MMTS, visited the site on September 7, 2024. During the site visit collar locations at Blueberry, Bend, Gulley and the Scottie Mine deposits were validated. The core storage site in Stewart was visited, with core from each deposit examined for mineralization and seven samples for re-assay obtained for validation of previous assay results. The QP is of the opinion that the data provided and used in the resource estimate for the Scottie Gold Mine Project deposits is adequate for resource estimation. There are no additional limitations to the exploration database for use in resource modelling.

12.5.2 Mining/Scottie Underground

Damian Gregory, P.Eng. of Datamine Canada Inc. (Snowden Optiro), visited the Property on July 29, 2025, and conducted a general project site overview in the proposed open pit, rock storage and underground mine portal areas.

It is the opinion of the QP that the underground as-built at Scottie must be confirmed prior to advancing the Project. As required under the **Health, Safey, and Reclamation Code for Mines in British Columbia, 6.25.4 Breakthrough to Mine Workings**:

No work shall be carried out within 30 m of abandoned or old workings, or any accumulation of water or unconsolidated material, or any other substance that may flow, unless the proposed work procedure has been approved by the manager.

The current as-built provided by Scottie Resources was digitized and extended by approximately 15 m using available historical plan-view maps. However, it may not accurately represent the actual development or mined-out stopes. Discussions with previous operators indicated that the underground mine used the shrinkage mining method, typically producing stopes about 5 – 10 ft. (4.5 m) wide. At the time of this report, Scottie Resources was in the process of engaging a firm to survey accessible areas of the historic Scottie underground mine.

Before advancing the Project, Snowden Optiro recommends that the historic as-built be reviewed and updated through an underground survey in all accessible areas. Where access is not possible, a 30 m offset should be maintained until probe drilling is conducted and/or a manager-approved work procedure is implemented to safely access and backfill the void.

12.5.3 Process and Infrastructure

Hassan Ghaffari, QP, visited the Scottie project site on July 29, 2025, inspecting the overall site infrastructure and general site conditions. It is the QP’s opinion that the overall site condition including access road is suitable for the proposed site infrastructures used in this technical report.

12.5.4 Metallurgy and Process

Dr. Jianhui (John) Huang, Ph.D., P.Eng., QP from Tetra Tech involved in the metallurgical test work review, gold recovery projections, preliminary process design, associated infrastructure and estimated operating costs, excluding mining related operating costs. He also visited Sepro Metallurgical Laboratory on August 27, 2025, to inspect the sample tested and laboratory facility and witnessed the DMS test sample preparation. The QP is of the opinion that the level of the test work completed so far is sufficient for the PEA study. The procedures used for the metallurgical tests are acceptable based on industrial procedures. The metallurgical performance projection was based on the test results, proposed mine plan and process plant design. The projections are considered to be reasonable and acceptable.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Since 1980, numerous phases of metallurgical testing have been conducted on samples from the Scottie Gold Mine Project to investigate gravity-recoverable gold content, whole-ore cyanide leaching performance, and metallurgical response of gravity tailings to flotation and cyanidation processes. These early studies laid the groundwork for understanding the mineralization's behavior under conventional recovery methods.

Beginning in 2024, the focus of metallurgical work shifted toward ROM pre-concentration using ore-sorting and dense media separation (DMS).

In this section, the test work is presented into four main sections as follows:

- Latest metallurgical ore-sorting test work conducted during 2024/2025
- Latest metallurgical pre-concentration DMS test work conducted during 2025
- 2025 head characterization test work
- 2023 Test Work on Conventional Recovery Methods
- Historical test work conducted to understand conventional recovery methods and mineralogy (1981 to 1994)

13.1 Ore Sorting Test Work 2025

Scottie Resources commissioned ABH Engineering Inc. in December 2024 to conduct an ore-sorting test program on a total of 210 individual quarter-core samples, with 70 samples collected from each of the following mineralization zones: Blueberry Open Pit (BBOP), Blueberry Underground (BBUG), and Scottie Gold Mine Underground (SGMUG). The quarter-core samples used were approximately 3" in size collected by breaking available drill core representing material with a wide gold grade range from each zone.

This ore-sorting test program was aimed at evaluating the amenability of the material to be separated into high grade pieces (concentrate) and low-grade pieces (waste) using X-ray scanning. The results from the test program are presented in the form of multiple operating parameters showing the feed grade, mass pull to product, gold recovery, sorted product grade, rejected waste grade, and upgrade ratio.

The samples were first scanned at ABH Engineering using an X-ray fluorescence (XRF) scanner. The samples were shipped and scanned by X-ray transmittance (XRT) at Saskatchewan Research Council (SRC) using a commercial-scale TOMRA XRT Tertiary sorter. The samples were then submitted for individual assays at SGS analytical labs in Burnaby. The assays and scan data were used to estimate sorter performance.

Below are the summarized test procedures and results from the XRT and XRF scanning test programs.

13.1.1 XRF Sorting Test Work

ABH Engineering in Surrey, BC received 210 quarter-core samples. The samples were scanned by ABH Engineering using a handheld XRF scanner. Each particle was scanned three times from different sides and orientations and the resulting XRF elemental readings were averaged to provide an elemental composition reading representative of the entirety of each rock.

13.1.1.1 XRF - Blueberry Open Pit (BBOP) Zone

To better understand the results from the test work, ideal grade-recovery curves (shown in green in Figure 13-1) are generated to show the theoretical maximum recoveries that can be achieved. It is impossible to achieve this ideal sorting curve, but it acts as a visualization aid and enables comparison of other algorithms with the ideal case.

For the BBOP zone, a multi-element XRF algorithm was developed using larger weights for Arsenic, Copper, Vanadium and a smaller weight for Iron due to the significant association of gold with these elements.

Figure 13-1 shows the XRF Grade-Recovery curve and how it compares to the ideal recovery curve. The XRF scans were mostly successful in predicting the grade of each particle. Although the data showed some variability; the recovery is close to the ideal especially for the first 40% mass rejection (reading the X-axis from right to left).

The blue vertical bars shown in the graph represent the assayed Au grade for each particle, and the order in which they are presented is dictated by what the XRF scans predicted to be the highest Au grade from the left to the right of the graph. As seen in the graph, the XRF algorithm is not capable of predicting the highest-grade material to lowest in perfect order, but it is effectively able to sort the high-grade material into the left side and the waste material to the right side. Additionally, the Au sorter recovery curve (shown in red) follows a similar trajectory to the ideal curve.

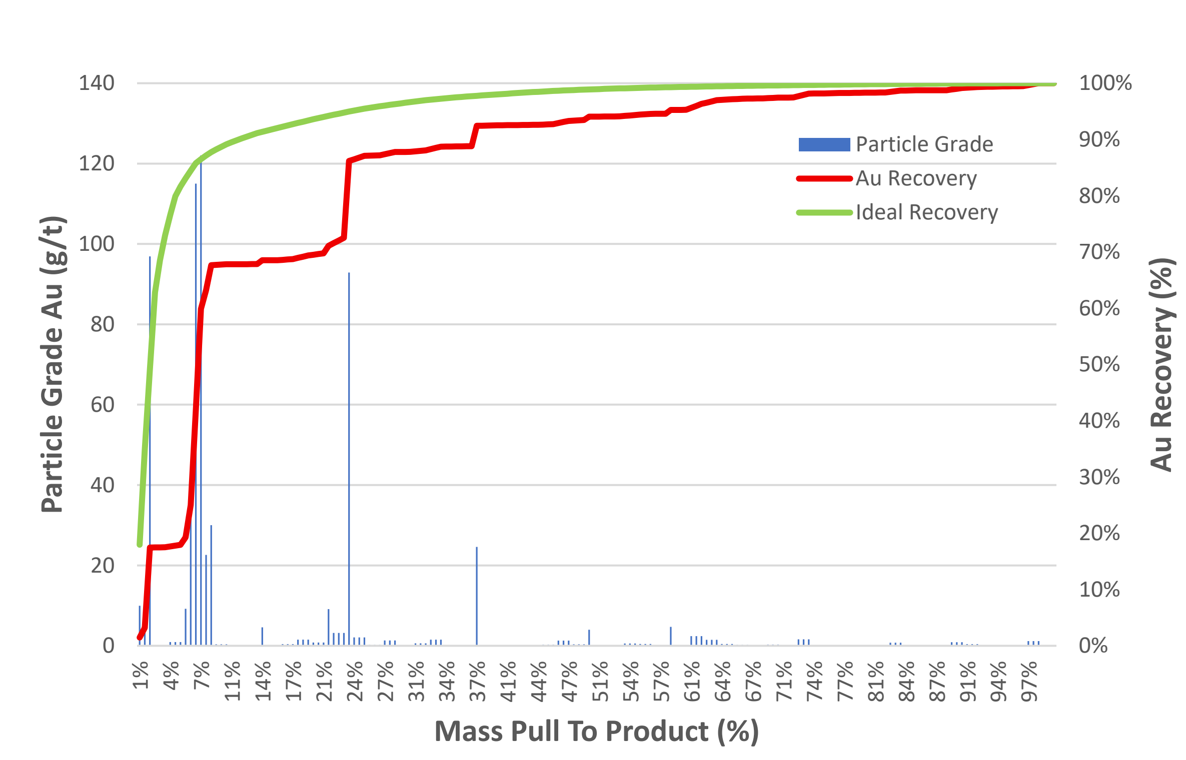


Figure 13-1: Grade Recovery Curve for XRF Sorting – BBOP Zone

Table 13-1 shows a summary of the operating points based on the XRF sorting algorithm for the BBOP zone. The table shows the mass pulls, sorter gold recoveries, sorter accepts and rejects grades, and upgrade ratios. Results show that the BBOP material, at an average grade of 3.8 g/t Au, can achieve a 2.3 to 4 times upgrade while recovering approximately 70% to 93% of the gold. The rejected material grade in this operating range is between 0.5 and 1.4 g/t Au.

Table 13-1: Summary of XRF Sorting Operating Points for the BBOP Zone (based on Assays and XRF Scans)

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
10%	68%	25.6	1.3	6.78
15%	69%	17.2	1.4	4.57
20%	70%	13.1	1.4	3.48
25%	87%	13.1	0.6	3.48
30%	88%	11.0	0.7	2.93
35%	89%	9.6	0.7	2.54
40%	93%	8.7	0.5	2.31
45%	93%	7.8	0.5	2.06
50%	94%	7.1	0.4	1.88

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
55%	94%	6.5	0.5	1.72
60%	95%	6.0	0.4	1.59
65%	97%	5.6	0.3	1.49
70%	97%	5.3	0.3	1.39
75%	98%	4.9	0.3	1.31
80%	98%	4.6	0.3	1.23
85%	99%	4.4	0.3	1.16
90%	99%	4.2	0.3	1.10
95%	99%	4.0	0.4	1.05

13.1.1.2 XRF - Blueberry Underground (BBUG) Zone

A multi-element XRF algorithm for the BBUG samples was developed, applying greater weighting to arsenic, copper, and reduced weighting for iron. The average grade of the samples (7.6 g/t Au) was close to the expected grade of the BBUG material (6.5 g/t Au).

Figure 13-2 shows the XRF Grade-Recovery curve and how it compares to the ideal recovery curve. The curve shows a very good sort, with recovery based on XRF sorting close to the theoretical maximum sorting at 50% or lower waste mass rejection.

Table 13-2 shows a summary of the operating points based on the XRF sorting algorithm for the BBUG zone. Results show that the BBUG material, at an average grade of 7.6 g/t Au, can achieve a 1.8 to 2.5 upgrade ratio while recovering approximately between 75% to 98% of the gold, in addition to the sharp separation of accept and waste at the 50% mass pull mark.

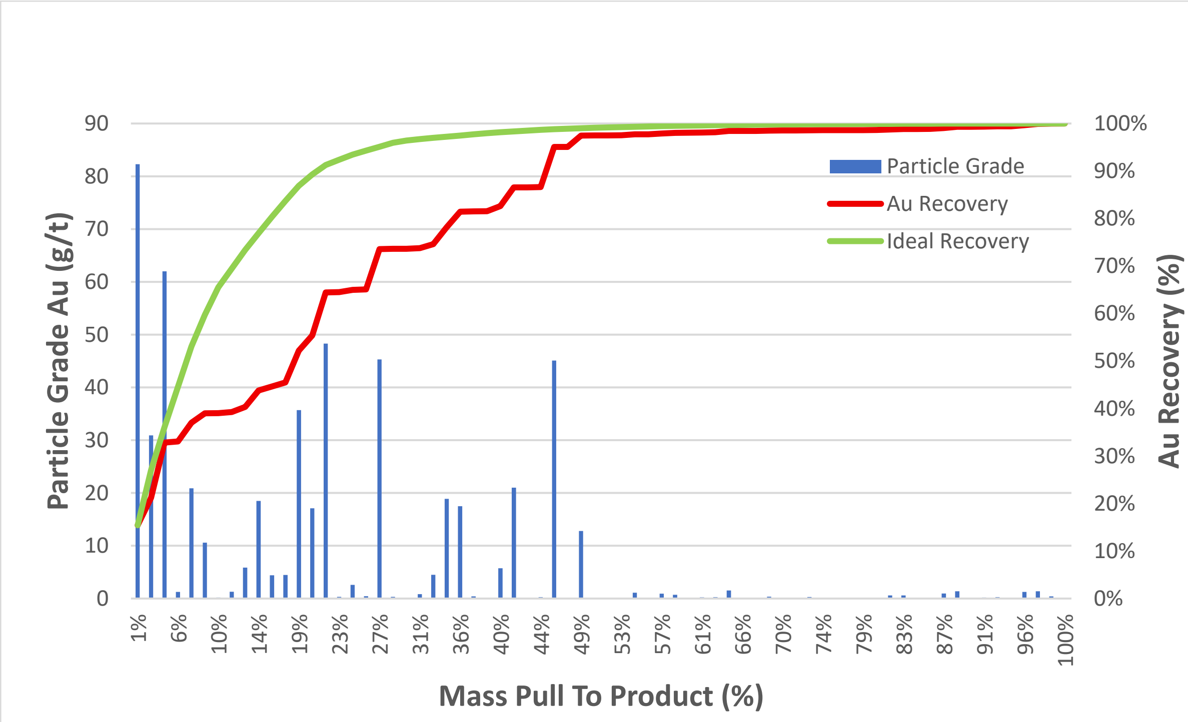


Figure 13-2: Grade-Recovery Curve for XRF Sorting - BBUG Zone

Table 13-2: Summary of XRF Sorting Operating Points for the BBUG Zone (based on Assays and XRF Scans)

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
4%	33%	58.4	5.3	7.66
10%	39%	29.7	5.2	3.90
14%	44%	23.4	5.0	3.07
20%	55%	21.1	4.3	2.77
24%	65%	20.4	3.5	2.68
30%	74%	18.7	2.9	2.45
34%	78%	17.4	2.5	2.28
40%	83%	15.7	2.2	2.07
44%	87%	14.9	1.8	1.96
50%	97%	14.9	0.4	1.95
54%	98%	13.7	0.4	1.80
60%	98%	12.5	0.4	1.63

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
64%	98%	11.7	0.3	1.53
70%	99%	10.7	0.4	1.41
74%	99%	10.1	0.4	1.33
80%	99%	9.4	0.5	1.23
84%	99%	8.9	0.6	1.17
90%	99%	8.4	0.5	1.10
94%	99%	8.0	0.8	1.05

13.1.1.3 XRF – Scottie Gold Mine Underground (SGMUG) Zone

A multi-element XRF algorithm for the SGMUG material was developed, applying greater weighting to arsenic and copper, and reduced weighting to iron. The average grade (6.9 g/t Au) of the SGMUG samples was lower than the expected grade of SGMUG material (10 g/t Au); however, it was considered to be close without the need for statistical modifications for the purpose of this test work.

Figure 13-3 shows the XRF Grade-Recovery curve for the SGMUG material and how it compares to the ideal recovery curve. The curve shows a very good sort and an effective separation when rejecting as much as 75% of the material. Additionally, between 45% and 75% mass rejection, the recoveries using the XRF scans for the SGMUG material are very close to the ideal recoveries.

Table 13-3 shows a summary of the operating points based on the XRF sorting algorithm for the SGMUG zone. Results show that the BBUG material, at an average feed grade of 6.9 g/t Au, can achieve a 1.8 to 3.5 upgrade ratio while recovering approximately between 86% to 98% of the gold, in addition to the sharp separation of accept and waste at the 55% mass pull mark. The rejected material grade for this operating range varied between 0.3 to 1.3 g/t Au.

The average grade of the Blueberry Underground samples of 7.6 g/t Au was considered close to the expected average grade of the material in this zone of 6.5 g/t Au for the purposes of this test program. This is also the case for the Scottie Gold Mine Underground samples with an average grade 6.9 g/t Au which was considered sufficiently close to the expected average grade of the material in this zone of 10 g/t Au for the purpose of this test program.

The average grade of the Blueberry Open Pit samples of 8.9 g/t Au was significantly higher than the expected average grade of the material in this zone of 3.5 g/t Au. To adjust for grade, modifications were made to the statistical distribution of the sample material, particularly the low-grade and waste, in order to match the expected BBOP zone average grade.

Results from the test program show that the sorting is highly effective for all the tested zones using either XRT or XRF technologies. XRF showed an edge over XRT for all tested zones in terms of mass pull to recovery ratio, particularly at operating mass pull to product range between 30 to 60%, with the added benefit of being able to conduct a three-way sort and create a low-grade reject simultaneously in one stage using commercially available XRF sorting machines.

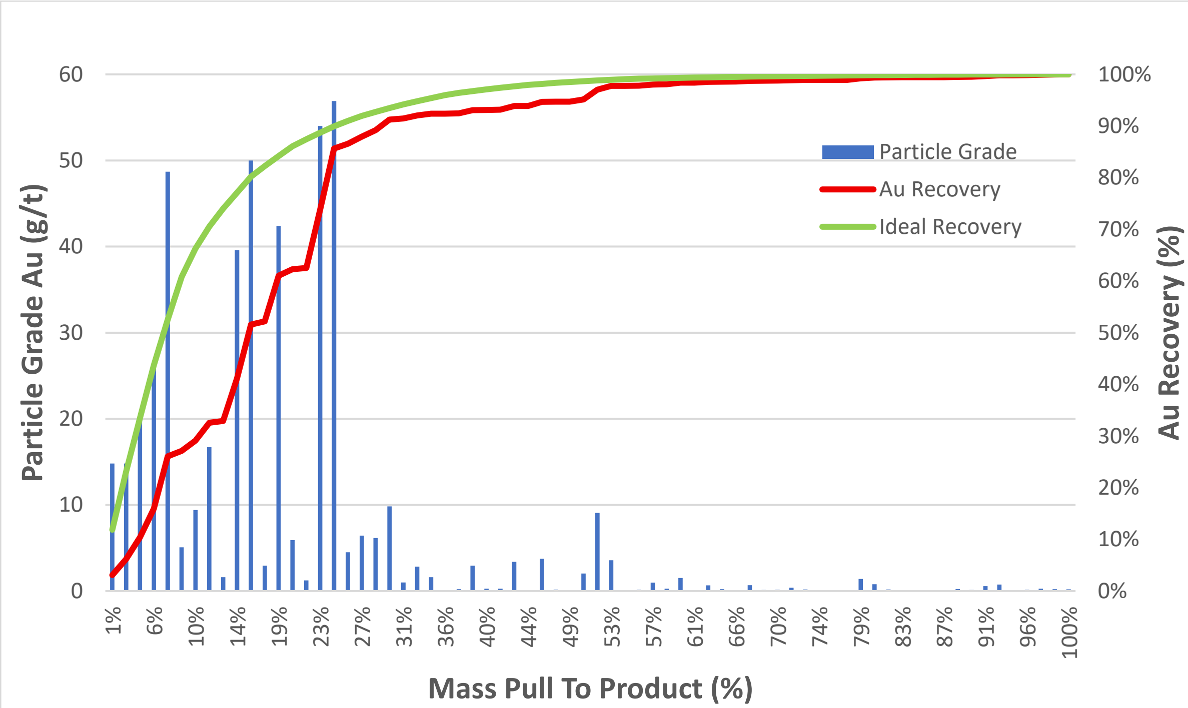


Figure 13-3: Grade-Recovery Curve for XRF Sorting - SGMUG Zone

Table 13-3: Summary of XRF Sorting Operating Points for the SGMUG Zone (based on Assays and XRF Scans)

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
4%	10%	16.7	6.4	2.43
10%	29%	19.9	5.4	2.91
14%	41%	19.7	4.7	2.88
20%	62%	21.3	3.2	3.11
24%	86%	24.2	1.3	3.53
30%	91%	20.8	0.9	3.04
34%	92%	18.5	0.8	2.69
40%	93%	16.0	0.8	2.33
44%	94%	14.5	0.8	2.12
50%	95%	13.0	0.7	1.90
54%	98%	12.3	0.3	1.80
60%	98%	11.2	0.3	1.64

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
64%	99%	10.5	0.3	1.53
70%	99%	9.7	0.3	1.41
74%	99%	9.1	0.3	1.33
80%	99%	8.5	0.2	1.24
84%	99%	8.1	0.2	1.18
90%	100%	7.6	0.3	1.11
94%	100%	7.3	0.2	1.06

13.1.2 X-Ray Transmittance (XRT) Sorting Test Work

After the XRF scans at ABH Engineering, the 210 samples were sent to Saskatchewan Research Council (SRC) and scanned using a full-scale TOMRA COM Tertiary sorter. All particles were marked with zone and sample number, and all were scanned by the machine at the same time.

From the X-ray images, the pixels of each particle are classified into seven categories according to the density readings, where low density is Density 0 and high density is Density 5 and Inclusions. Figure 13-4 shows the colour legends for the different density classes and Figure 13-5 shows the XRT processed scans for all the samples at SRC.

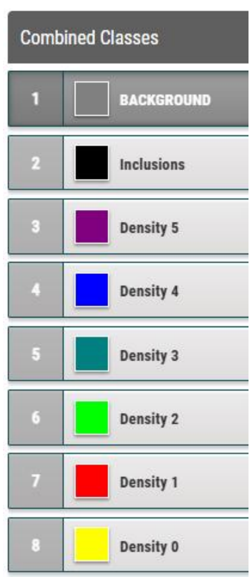


Figure 13-4: XRT Density Reading Legends (2: Highest Density, 8: Lowest Density)

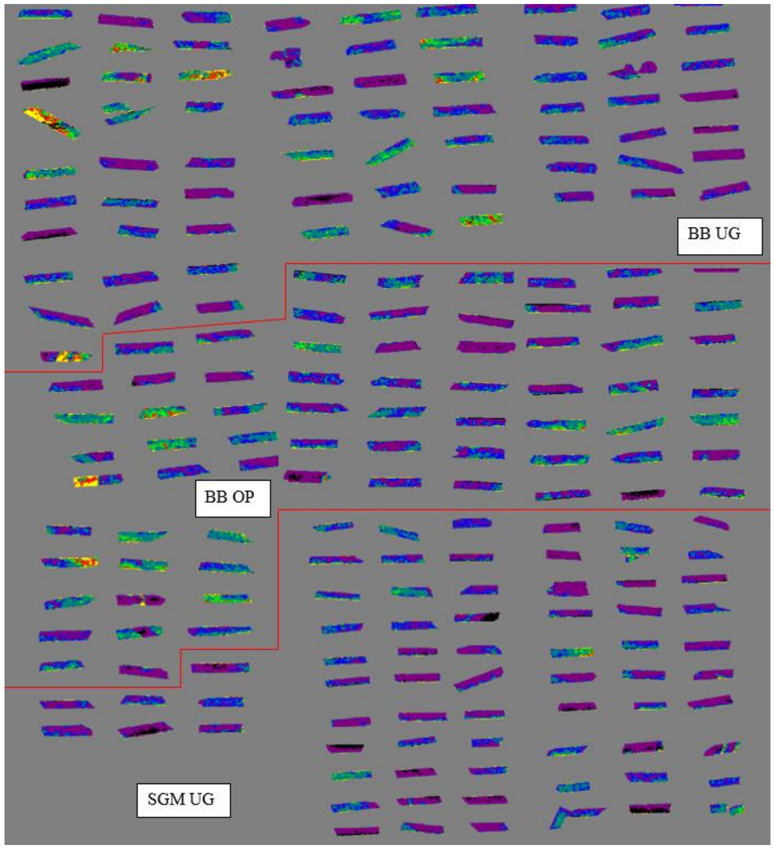


Figure 13-5: XRT Scans Showing the Variation in Densities Between Particles

These XRT scans were analyzed with the assay results corresponding to each sample to determine the correlations. An algorithm that uses a positive correlation for higher-density pixels, and a negative correlation for lower-density pixels was developed that gives a prediction for the possible grade of each particle. The particles were then sorted in descending order according to a combination of those weights created by the algorithm from which the XRT Grade-Recovery curves were created for each zone.

13.1.2.1 XRT - Blueberry Open Pit (BBOP) Zone

Similar to the graphs shown for the XRF analysis Figure 13-6 shows the XRT Grade-Recovery curve for the BBOP zone and how it compares to the ideal recovery curve. The XRT recovery was very close to the ideal curve for the first 30% mass rejection, then slightly diverging between 30% and 70% mass rejection.

Table 13-4 below shows the different operating points of the XRT sorting algorithm for the BBOP sample material with an average feed grade of 3.8 g/t Au. An upgrade between 1.4 and 2.8 can be achieved while recovering approximately 85% to 95% of the gold. The rejected material grade in this operating range was approximately 0.8 g/t Au.

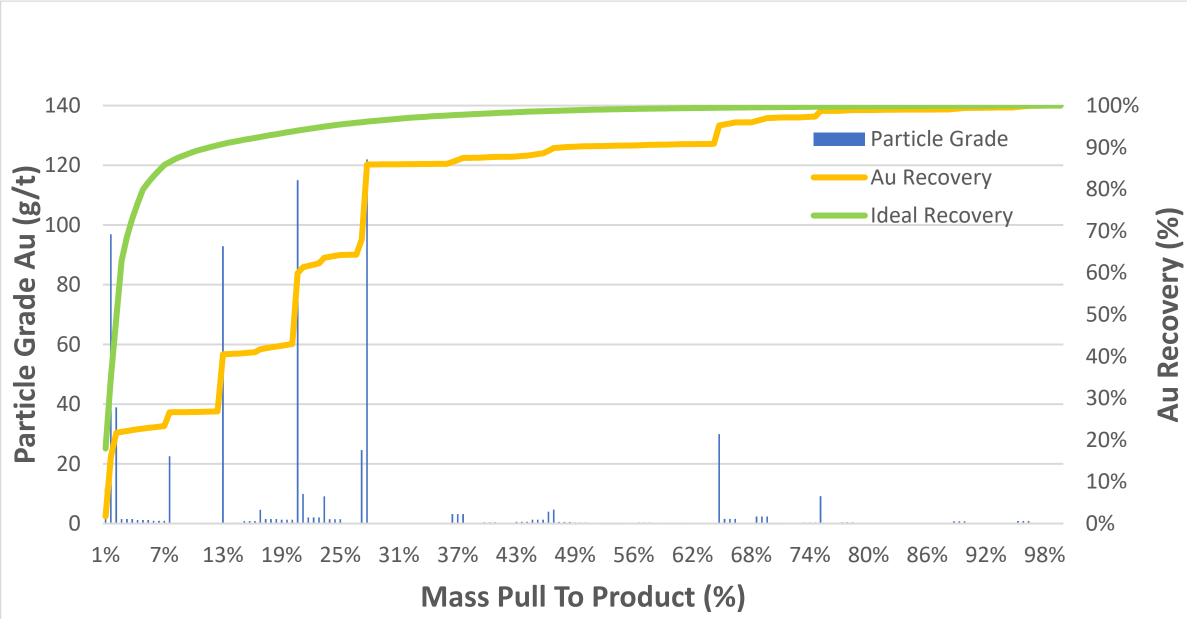


Figure 13-6: Grade-Recovery Curve for XRT Sorting – BBOP Zone

Table 13-4: Summary of XRT Sorting Operating Points for the BBOP Zone (based on Assays and XRT Scans)

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
10%	27%	10.1	3.1	2.67
15%	41%	10.3	2.6	2.72
20%	43%	8.1	2.7	2.15
25%	64%	9.7	1.8	2.57
30%	86%	10.8	0.8	2.86
35%	86%	9.3	0.8	2.46
40%	88%	8.3	0.8	2.19
45%	88%	7.4	0.8	1.96
50%	90%	6.8	0.7	1.80
55%	90%	6.2	0.8	1.64
60%	91%	5.7	0.9	1.51
65%	95%	5.5	0.5	1.47
70%	97%	5.2	0.4	1.39
75%	99%	5.0	0.2	1.32

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
80%	99%	4.7	0.2	1.24
85%	99%	4.4	0.2	1.16
90%	99%	4.2	0.2	1.10
95%	100%	4.0	0.3	1.05

13.1.2.2 XRT - Blueberry Underground (BBUG) Zone

Figure 13-7 shows the XRT Grade-Recovery curve for the BBUG zone, and how it compares to the ideal recovery curve. The XRT curve is close to the ideal curve for the first 60% of the rejected mass and XRT is able to effectively separate the high gold grade particles from the waste for this operating range.

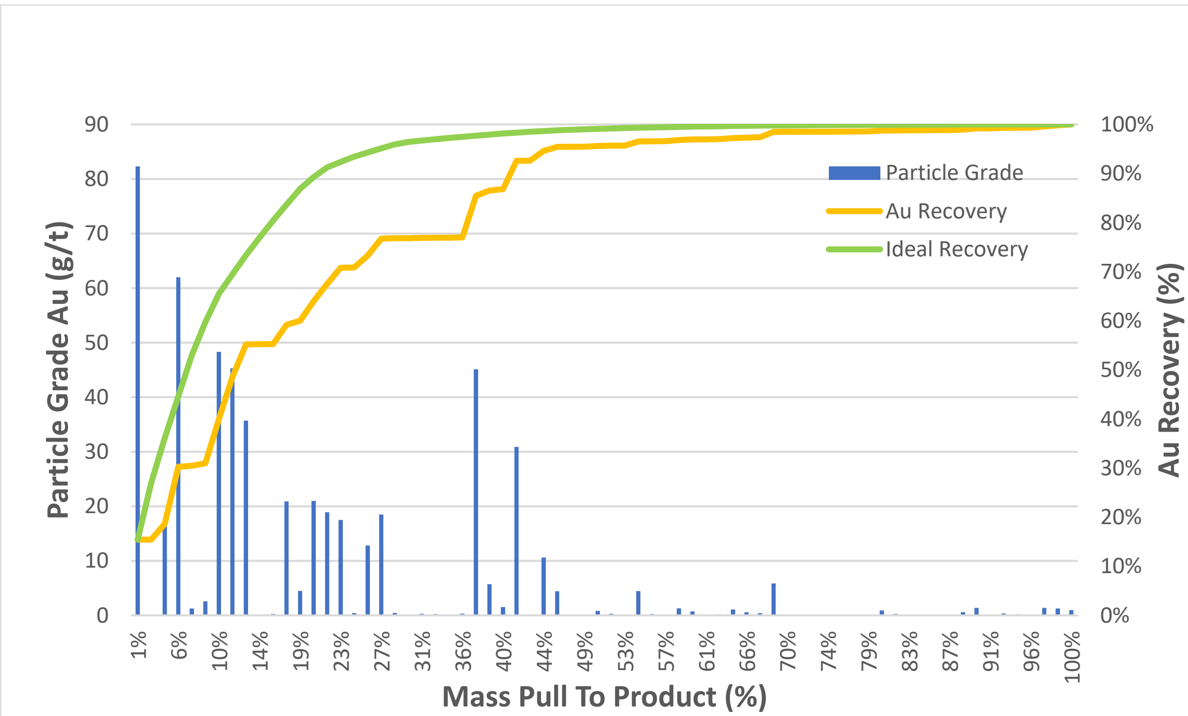


Figure 13-7: Grade-Recovery Curve for XRT Sorting - BBUG Zone

Table 13-5 below shows the different operating points of the XRT sorting algorithm for the BBUG sample material with an average feed grade of 7.6 g/t Au. The material can be upgraded by a factor of 1.8 and 2.5 while recovering approximately 77% to 97% of the gold. The rejected material grade in this operating range varied between 0.6 and 2.5 g/t Au.

Table 13-5: Summary of XRT Sorting Operating Points for the BBUG Zone (based on Assays and XRT Scans)

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
4%	19%	33.1	6.5	4.35
10%	40%	30.5	5.1	4.00
14%	55%	29.5	4.0	3.87
20%	64%	24.4	3.4	3.20
24%	71%	22.2	2.9	2.92
30%	77%	19.5	2.5	2.56
34%	77%	17.1	2.7	2.24
40%	87%	16.5	1.7	2.17
44%	95%	16.3	0.7	2.14
50%	96%	14.6	0.7	1.91
54%	97%	13.6	0.6	1.78
60%	97%	12.3	0.6	1.62
64%	97%	11.5	0.6	1.51
70%	99%	10.7	0.4	1.41
74%	99%	10.1	0.4	1.33
80%	99%	9.4	0.5	1.23
84%	99%	8.9	0.6	1.17
90%	99%	8.4	0.6	1.10
94%	99%	8.0	0.9	1.05

13.1.2.3 XRT – Scottie Gold Mine Underground (SGMUG) Zone

Similar to the XRF results for the SGMUG samples, the XRT algorithm is able to effectively separate the high-grade material from waste as seen in Figure 13-8. The XRT sorting recovery curve is almost identical to the ideal curve for the first 45% mass rejection and follows a similar trajectory for the additional 30% rejected material implying that significant high grading can be achieved on this material.

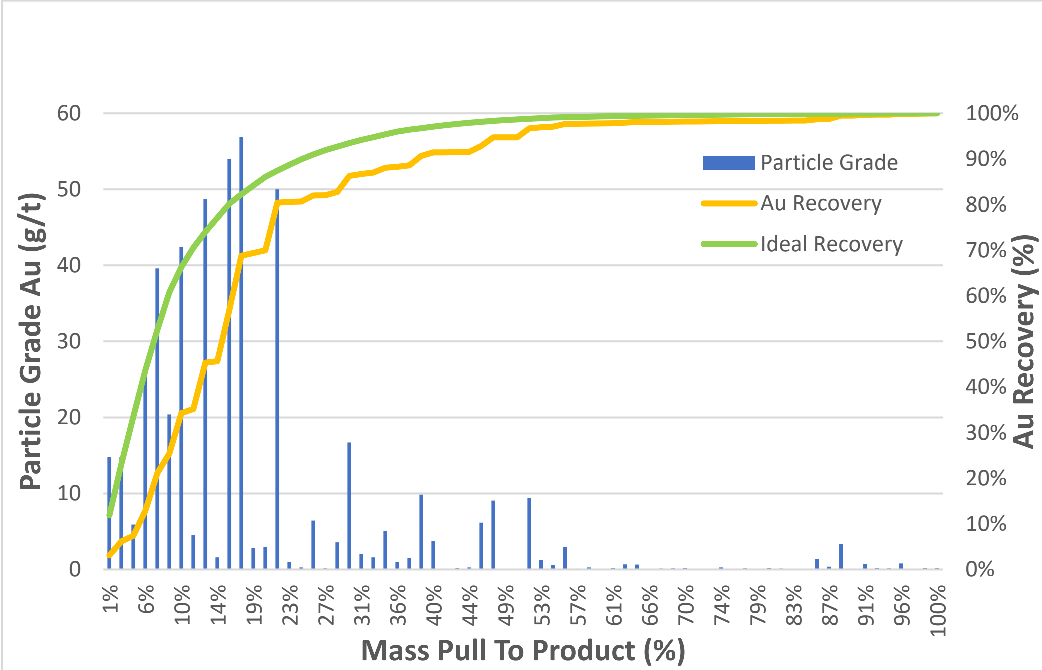


Figure 13-8: Grade-Recovery Curve for XRT Sorting - SGMUG Zone

Table 13-6 below shows the operating conditions generated based on the XRT algorithm for the SGMUG material. Results show that it is feasible to achieve a 1.8 to more than 3 times upgrade while recovering 80% to 97% of the gold with a significant mass rejection. The rejected material in this operating range varied between 0.5 and 1.7 g/t Au.

Table 13-6: Summary of XRT Sorting Operating Points for the SGMUG Zone (based on Assays and XRT Scans)

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
4%	7%	11.8	6.6	1.73
10%	34%	23.5	5.0	3.42
14%	46%	21.9	4.3	3.20
20%	70%	24.0	2.6	3.50
24%	81%	22.8	1.7	3.32
30%	86%	19.7	1.3	2.88
34%	88%	17.6	1.2	2.57
40%	91%	15.7	1.0	2.29
44%	92%	14.2	1.0	2.07
50%	95%	13.0	0.7	1.89

Mass Pull (%)	Recovery (%)	Accept Grade (g/t Au)	Reject Grade (g/t Au)	Sorter Au Upgrade Ratio
54%	97%	12.3	0.4	1.79
60%	98%	11.2	0.4	1.63
64%	98%	10.5	0.4	1.53
70%	98%	9.6	0.4	1.40
74%	98%	9.1	0.5	1.32
80%	98%	8.4	0.6	1.23
84%	98%	8.0	0.7	1.17
90%	100%	7.6	0.3	1.11
94%	100%	7.2	0.3	1.06

13.1.3 Comparison of XRF & XRT Results and Conclusions

The recovery curves for XRT, XRF, and Ideal cases were plotted on the same graph in order to compare the technologies and assist in assessing the better technology to move forward with for the next phase of testing. Figure 13-9 shows the comparison for the BBOP zone using the grade corrected data, Figure 13-10 for the BBUG zone and Figure 13-11 for the SGMUG zone.

For the first zone Blueberry Open Pit, XRF shows a slight but consistent advantage in recovery over XRT, with a major advantage at more aggressive (smaller) mass pulls to product. For the Blueberry Underground zone, XRF and XRT are very close and nearly tied in terms of recovery, with XRT showing a slight advantage at more aggressive mass pulls, and XRF at larger mass pulls to product. For the Scottie Gold Mine Underground zone XRT shows an advantage to XRF for aggressive mass pulls to product, while XRF shows an advantage at above 25% mass pulls to product. All technologies succeeded in sorting the material and achieved similar recoveries at different mass pulls, with XRF showing a slight advantage in aggregate and the optionality that it can achieve three-way sorting in a single stage to create a low-grade stockpile along with the high-grade plant feed.

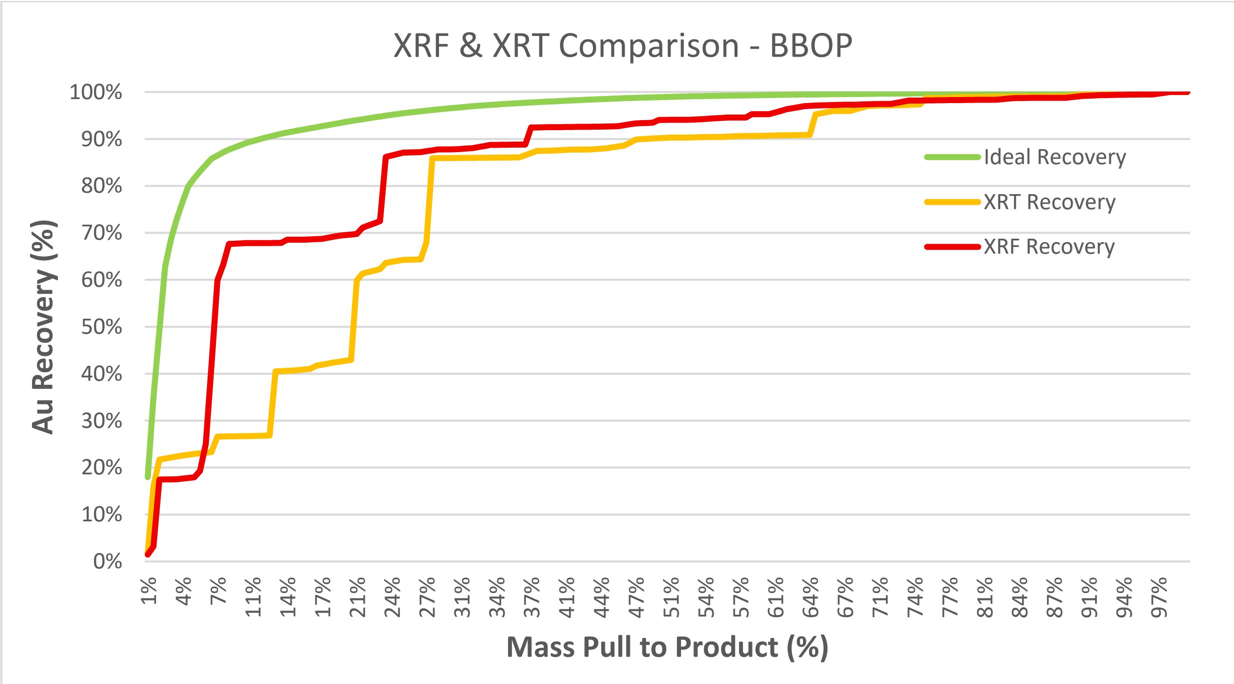


Figure 13-9: XRF and XRT Results Comparison - BBOP Zone

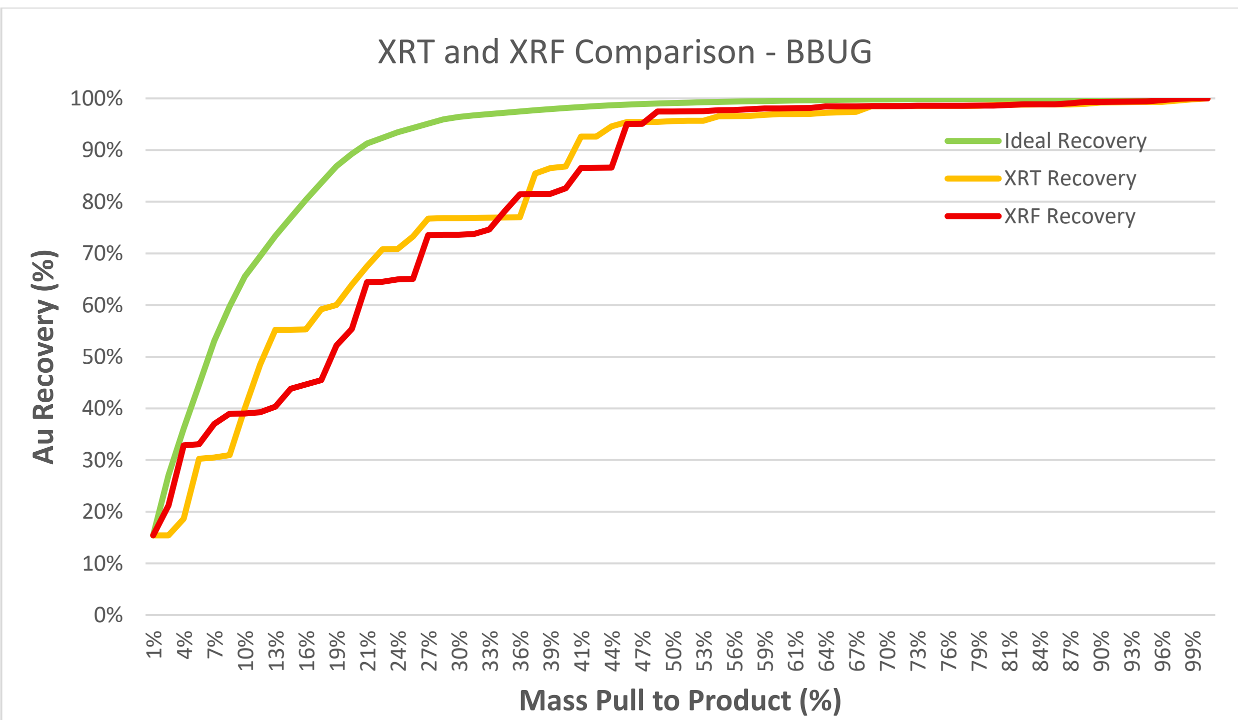


Figure 13-10: XRF and XRT Results Comparison - BBUG Zone

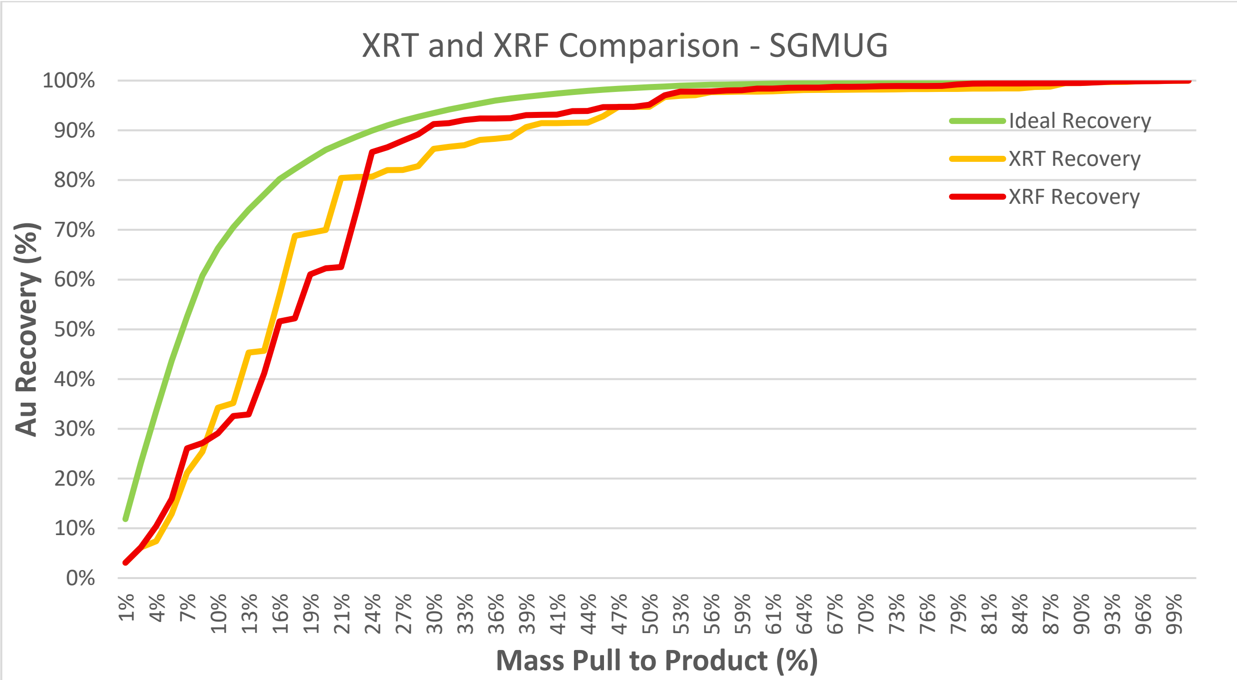


Figure 13-11: XRF and XRT Results Comparison - SGMUG Zone

13.2 Dense Media Separation (DMS) Test Work 2025

Heavy Liquid Separation (HLS) investigation was conducted to assess the mineralization's amenability to DMS and to determine the optimal specific gravity cut point for effective waste rock rejection.

13.2.1 Samples and Procedure

Approximately 15 to 17 kg of sample material was collected individually from each of the three zones: BBUG, BBOP, and SGMUG zones. Upon arrival, each sample was weighed, catalogued, and stage crushed to a top size of 19.05 mm (¾"). After homogenization, half of the crushed material from each zone was split and stored, while the remainder was further crushed to 6.7 mm (¼"). A 1 kg sub-sample from each zone was rotary split for head assay, and the remaining material was designated for HLS testing. The test charge was wet screened at 0.85 mm to remove fines, and the +0.85 mm fraction underwent HLS testing at four specific gravity (SG) cut points. All HLS products were then washed, dried, crushed, sub-sampled, pulverized, and submitted for Au fire assay, along with duplicate head samples for Au fire assay and multi-element ICP analysis.

The HLS procedure involved densiometric separation using baths of varying solution densities. For each SG cut point, the solution's specific gravity was adjusted before the +0.85 mm test charge was introduced and mixed. After allowing sufficient time for particle separation, floating fractions were collected, washed, and dried. Once separation at a given SG was complete, the solution was drained, and products were prepared for the next SG level. Depending on the SG direction, either the floats (for decreasing SG) or the sinks (for increasing SG) were reused in subsequent tests. This process was repeated through all designated SG cut points for each individual zone sample. A total of four (4) different SG cut-points of 3.05, 2.95, 2.85, and 2.75 were conducted for this HLS test program to generate data for the DMS process assessment. The test work was conducted using lithium meta tungstate as the heavy medium.

13.2.2 HLS Results

The HLS results on three different samples BBUG, BBOP, and SGMUG are presented in Table 13-7, Table 13-8, and Table 13-9 respectively.

For the BBUG sample with head grade of 10.01 g/t Au, at an SG cut point of 2.85, the sinks (fines to tailings) yielded an overall gold recovery of 59.6%, with a product grade of 16.68 g/t Au and a total mass yield of 35.8%. The BBUG sample exhibited a sufficiently high fines grade of 12.41 g/t Au, making it suitable for blending with the SG >2.85 preconcentrate. This combination enhanced the overall gold recovery to 88.6%, producing a concentrate with a grade of 14.99 g/t Au and a mass yield of 59.2%.

For the BBOP sample with head grade of 3.02 g/t Au, at an SG cut point of 2.95, the sinks yielded a gold recovery of 57.4%, with a concentrate grade of 15.00 g/t Au and a mass yield of 11.6%. However, the fines—grading at 3.02 g/t Au—are not suitable for blending with the HLS preconcentrates, as doing so results in a final concentrate grade below 15 g/t Au. To improve overall gold recovery for this sample, further upgrading of the fines is recommended.

For the SGMUG sample with head grade of 5.83 g/t Au, at an SG cut point of 2.95, an overall gold recovery of 52.7% was achieved from the sinks (fines to tailings), yielding a product grade of 15.78 g/t Au with a mass yield of 19.5%. However, blending the fines with the HLS preconcentrate results in a concentrate grade below 15 g/t Au across all SGs. To enhance gold recovery for this sample, further upgrading of the fines is recommended.

Table 13-7: HLS Results Summary for Blueberry Underground

Specific Gravity	Fines included with Tails			Fines included with Concentrate		
	Yield (%)	Au (g/t)	Recovery (%)	Yield (%)	Au (g/t)	Recovery (%)
3.05	10.7	31.15	33.4	34.2	18.3	62.4
2.95	17.1	23.63	40.3	40.5	17.14	69.3
2.85	35.8	16.68	59.6	59.2	14.99	88.6
2.75	60.9	11.46	69.7	84.3	11.72	98.7
Head		10.01			10.01	

Table 13-8: HLS Results Summary for Blueberry Open Pit

Specific Gravity	Fines included with Tails			Fines included with Concentrate		
	Yield (%)	Au (g/t)	Recovery (%)	Yield (%)	Au (g/t)	Recovery (%)
3.05	7.5	20.23	50.3	32.3	7.02	75.1
2.95	11.6	15	57.4	36.3	6.83	82.2
2.85	25.3	7.9	66.3	50.1	5.49	91.1
2.75	54.1	4.04	72.4	78.9	3.72	97.2
Head		3.02			3.02	

Table 13-9: HLS Results Summary for Scottie Gold Mine Underground

Specific Gravity	Fines included with Tails			Fines included with Concentrate		
	Yield (%)	Au (g/t)	Recovery (%)	Yield (%)	Au (g/t)	Recovery (%)
3.05	14.9	19.36	49.5	37.9	11.13	72.4
2.95	19.5	15.78	52.7	42.5	10.38	75.7
2.85	32.8	10.24	57.7	55.8	8.41	80.6
2.75	59.8	6.84	70.2	82.8	6.56	93.1
Head		5.83			5.83	

13.3 2025 Sample Head Characterization Test Program

Head characterization test program was conducted on one master composite sample from Scottie Resources. The program is designed to achieve two primary objectives: to evaluate gold deportment by size fraction assay and to conduct comminution testing to support engineering design, including Bond Ball Mill Work Index, Bond Rod Mill Work Index, Bond Low Impact Crushing Index, and Bond Abrasion Index. In addition, the program measured moisture content, bulk density, and specific gravity of the sample.

13.3.1 Size Fraction Assay

For this analysis, a 15 kg sample was crushed at 8 mm and screened into multiple size intervals (8 mm, 4 mm, 1.5 mm, 850 µm, 425 µm, 212 µm, 75 µm, and 38 µm). Fractions greater than 150 µm were analyzed by screen-grade assay to determine gold distribution in the tested size fractions, while finer fractions were assayed in duplicate using standard fire assay. The assay results are presented in Table 13-10. The size fraction assay indicates that gold distribution closely follows the mass distribution to each fraction, as the grades across the size fractions are relatively similar.

Table 13-10: Size by Size Fraction Assay Results

Size (µm)	Size Fraction Assay		
	Weight Fraction (%)	Gold Distribution (%)	Gold (g/t)
9500	21.22	17.4	10.71
4750	39.69	44.9	14.79
1700	19.64	20.1	13.38
850	6.09	5.7	12.33
425	4.89	4.5	12.07
212	2.56	2.4	12.16
75	2.72	2.4	11.56
38	1.26	1.2	12.18
-38	1.94	1.4	9.78
Net	100		13.08
Head 1			14.36
Head 2			12.92

13.3.2 Comminution Test

Approximately 20 pieces of the sample, sized between 75 mm and 55 mm and totaling about 10 kg, were supplied for the Bond Impact Crushing Index Test. In addition, 60 kg of the sample was provided for the remaining test work, with most of this material consisting of particles larger than 25 mm. The results of the Bond Ball Mill Work Index, Bond Rod Mill Work Index, Bond Impact Crushing Index, and Bond Abrasion Index tests are presented in Table 13-11. Based on the Bond ball mill work index result, the sample can be categorized as medium hard material.

Table 13-11: Comminution Characterization Test Results

Description	Unit	Value
Bond Low Impact Crusher Work Index, Average	kWh/tonne	11.46
Bond Rod Mill Work Index, BRWi	kWh/tonne	11.80
Bond Ball Mill Work Index, BBWi	kWh/tonne	10.98
Bond Abrasion Index, Ai		0.119

13.3.3 Other Physical Parameters

The moisture content, specific gravity, bulk density, and repose angle of the sample are presented in Table 13-12.

Table 13-12: Physical Parameters Results

Description	Unit	Value
Moisture Content, as received	%	0.54
Specific Gravity		3.47
Natural Bulk Density	g/cm^3	1.77
Compacted Bulk Density	g/cm^3	1.91
Angle of Repose, Funnel Method	degree	31.7
Angle of Repose, Tilting Box Method	degree	35.1

13.4 2023 Test Work on Conventional Recovery Methods

To evaluate the amenability for conventional processes, this test program was designed to evaluate the gravity recoverable gold (GRG) content, assess the cyanide leaching performance of the head composite sample, and examine the response of gravity tailings to both flotation and cyanidation processes.

13.4.1 Sample Preparation and Test Work Procedure

On January 3, 2023, Sepro Laboratories Inc. received approximately 260 kg of material from Scottie Resources, delivered in sixteen rice bags. The samples were combined to form a single composite for testing. Upon receipt, each sample was weighed, cataloged, and then combined and homogenized into a single composite sample as per client instructions. This composite was then split into representative test charges using a rotary splitter. Sub-samples were analyzed for gold and silver content via fire assay, underwent multi-element ICP scanning, and were tested for carbon speciation.

A Particle Size Analysis (PSA) was conducted to determine the size distribution of the composite material. A standard three-stage GRG test was performed on a 20 kg portion using a laboratory-scale Falcon L40 centrifugal concentrator.

Subsequent test work included rougher flotation tests on the GRG tailings at two grind sizes (80% passing (p80) of 103 µm and 55 µm), with additional tests using different reagent schemes at 103 µm. Cleaner flotation tests were also conducted, both with and without regrinding the rougher concentrate. Finally, cyanidation tests were carried out on the gravity tailings for gold and silver recovery, along with a cyanide leach kinetics test on the head composite sample at a grind size of 25µm. The overall test work flowsheet is presented in Figure 13-12.

13.4.2 GRG Procedure

GRG represents the maximum theoretical percentage of gold recoverable from a mineralization using gravity concentration methods. It involves staged grinding to liberate gold particles, followed by gravity separation to evaluate gravity concentration efficiency.

13.4.3 Flotation Procedure

Rougher and cleaner flotation tests were performed on a sub-sample of the GRG tailings under various conditions. Tests were conducted both with and without regrinding the tailings prior to flotation. The reagent suite included Potassium Amyl Xanthate (PAX) as the primary collector, Dialkyl dithiophosphate 3418A (a phosphorus-based collector), Aero Maxgold (AMG900) as a gold recovery promoter, Methyl Isobutyl Carbinol (MIBC) as a frother, and copper sulphate as an activator, all under natural pH conditions.

Two rougher flotation tests were carried out on the GRG tailings at a grind size of p80 103 μm—one without regrind and one with regrind to p80 55 μm. Two additional rougher tests were conducted at p80 103 μm using alternative reagent schemes. Cleaner flotation tests involved up to three cleaning stages using the same reagents, with dosages adjusted based on visual observations and intermediate results.

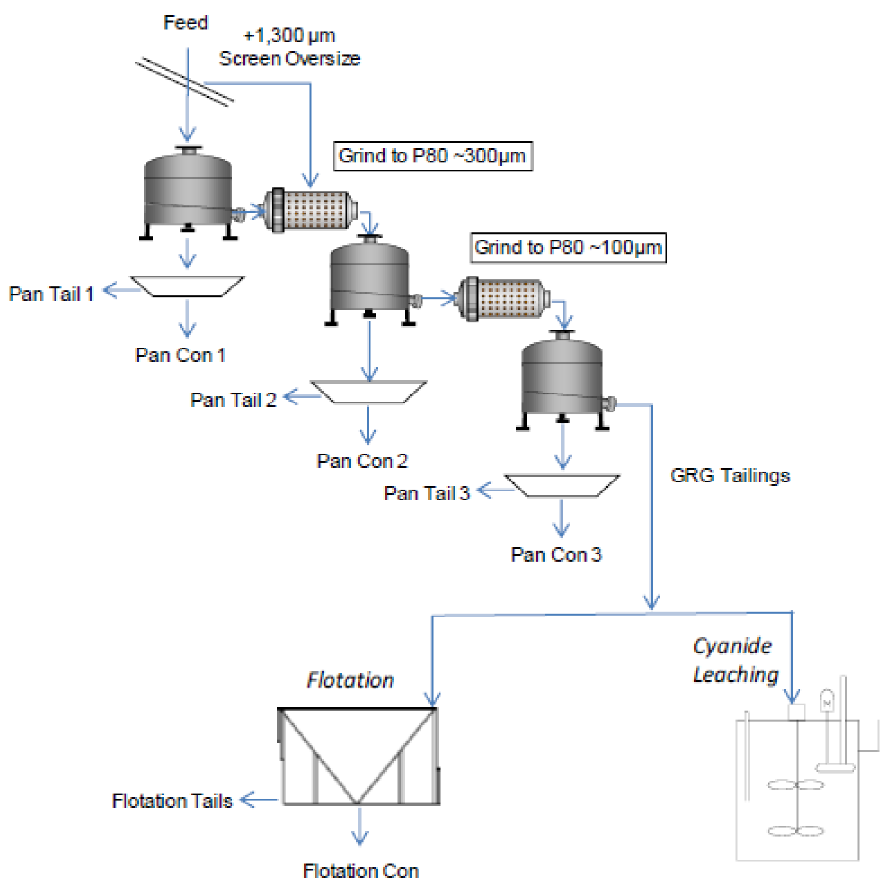


Figure 13-12: 2023 SEPRO Test Work Program Flowsheet

13.4.4 Cyanidation Procedure

A 72-hour bottle roll cyanide leach test was conducted on both the head composite sample and a sub-sample of the GRG tailings. The tests were performed at 40% solids and pre-conditioned with lime for one hour to stabilize the pH before cyanide addition. A cyanide concentration of 1 g/L NaCN was maintained, with pH kept above 10.0 throughout the test.

Solution subsamples were collected at intervals of 1, 3, 6, 24, 48, and 72 hours, along with measurements of cyanide concentration, dissolved oxygen, and pH. After 72 hours, pregnant leach solutions were sampled, the slurries were filtered, and the residues were washed in multiple stages. Two residue subsamples were submitted for assay.

13.4.5 Results and Discussion

13.4.5.1 Head Sample Characterization

The representative composite samples were analysed for gold, silver and multi-element and carbon speciation. Table 13-13 lists the results of fire assay and key multi-element analysis results.

Table 13-13: Head Sample Chemical Analysis Results

Element	Unit	Head	Duplicate Head	Head (Average)	Method
Au	ppm	15.9	7.7	11.8	Fire Assay
Ag	ppm	8	5	6.5	Fire Assay
As	%	0.044	0.049	0.047	Four-Acid ICP
Cu	%	0.05	0.06	0.06	Four-Acid ICP
Fe	%	10.45	11.13	10.79	Four-Acid ICP
S	%	3.74	4.12	3.93	Four-Acid ICP

The head gold grade of the sample, based on both direct fire assay and calculated results from cyanide leach and GRG tests, ranged from 7.7 to 15.9 g/t Au, with an average of 11.1 g/t Au, as presented in Table 13-14. Variability in the results is attributed to the “nugget effect,” common in samples with high GRG content. To reduce this effect, Sepro favors the calculated head grade, which reflects assays from multiple products for a more representative value.

Table 13-14: Gold Head Sample Grade

Sample	Au (g/t)
Head Sample	15.9
Duplicate Head Sample	7.7

Sample	Au (g/t)
GRG, Calculated	9.4
CN, Calculated	11.4
Average	11.1

The carbon speciation results are presented in Table 13-15. The presence of organic carbon is expected to have the negative potential on gold recovery due to pre-robbing effect.

Table 13-15: Carbon Speciation on Head Sample

Sample	Unit	Total C	Total C, Inorg	Total C, Org	Graphitic C
Head	%	1.16	0.9	0.25	<0.02
Head (Duplicate)	%	1.15	0.84	0.3	<0.02
Head (Average)	%	1.16	0.87	0.28	<0.02

13.4.5.2 GRG Results

The GRG test results are given in Table 13-16. The composite sample was determined to have a GRG value of 37.6%. Grinding of the material to a p80 of 103 µm was beneficial as approximately 14.6% of the GRG was recovered after the final grind. The elevated pan concentrate grades indicate the centrifugal gravity concentrate is amenable to further upgrading by a secondary gravity process, such as a shaking table.

Table 13-16: GRG Test Results

Grind Size (P80 in µm)	Products	Weight	Au		Ag	
	(g)	(%)	g/t	Distribution %	g/t	Distribution %
1885	Pan Conc 1	0.08	515.0	4.4	140.0	1.5
	Pan Tails 1	0.64	70.2	4.8	84.0	7.0
	L40 Conc 1	0.72	120.1	9.2	90.3	8.5
300	Pan Conc 2	0.06	1102.7	7.3	376.0	3.1
	Pan Tails 2	0.45	135.5	6.6	120.0	7.2
	L40 Conc 2	0.52	251.6	13.8	150.7	10.2
103	Pan Conc 3	0.05	955.7	5.6	418.0	3.0
	Pan Tails 3	0.46	185.9	9.0	120.0	7.2
	L40 Conc 3	0.51	268.2	14.6	151.9	10.2

Grind Size (P80 in µm)	Products (g)	Weight (%)	Au		Ag	
			g/t	Distribution %	g/t	Distribution %
103	Total L40 Conc	1.75	202.3	37.6	126.2	29.0
	L40 Tails	98.25	6.0	62.4	5.5	71.0

13.4.5.3 Flotation Test Results

Initial rougher flotation tests were conducted on the standard GRG tailings (Test ZR102) to evaluate the response of gravity tailings to flotation using $CuSO_4$ and PAX, and to compare performance at different grind sizes. Test ZR202 was performed without grinding (p80 of 103 µm), while test ZR203 included additional grinding to a p80 of 55 µm.

To assess the impact of reagent variations, two additional flotation tests were carried out at p80 of 103 µm: ZR204 using CuS $_4$, PAX, and 3418A, and ZR205 using $CuSO_4$, PAX, 3418A, and AMG900. Results from these rougher flotation kinetics tests are shown in Figure 13-13.

Test ZR202 achieved a gold recovery of 71.4% with a concentrate grade of 32.4 g/t Au and a mass yield of 12.9%. At the finer grind (Test ZR203), gold recovery increased to 79.8%, though the grade decreased to 26.8 g/t Au with a higher mass yield of 18.3%. Gold flotation kinetics were notably faster at p80 of 55 µm, reaching 69% recovery within the first 3 minutes. Extending flotation time beyond 5 minutes primarily increased mass pull with minimal gains in gold recovery. The addition of 3418A and AMG900 (Tests ZR204 and ZR205) did not enhance gold kinetics.

Gold losses to flotation tailings were lower in the finer grind test (ZR203), with 20.2% of gold reporting to tailings. Silver flotation kinetics remained consistent across grind sizes and with 3418A, but declined with the use of AMG900. A direct correlation was observed between gold, silver, and sulphur recoveries, indicating the presence of sulphide minerals in the flotation concentrates. Notably, the rougher concentrate from Test ZR203 contained 0.5% Zn with a recovery of 92%.

Gold losses to tailings are likely due to incomplete liberation or gold locked within gangue minerals, primarily silicates, although no mineralogical analysis was performed to confirm this.

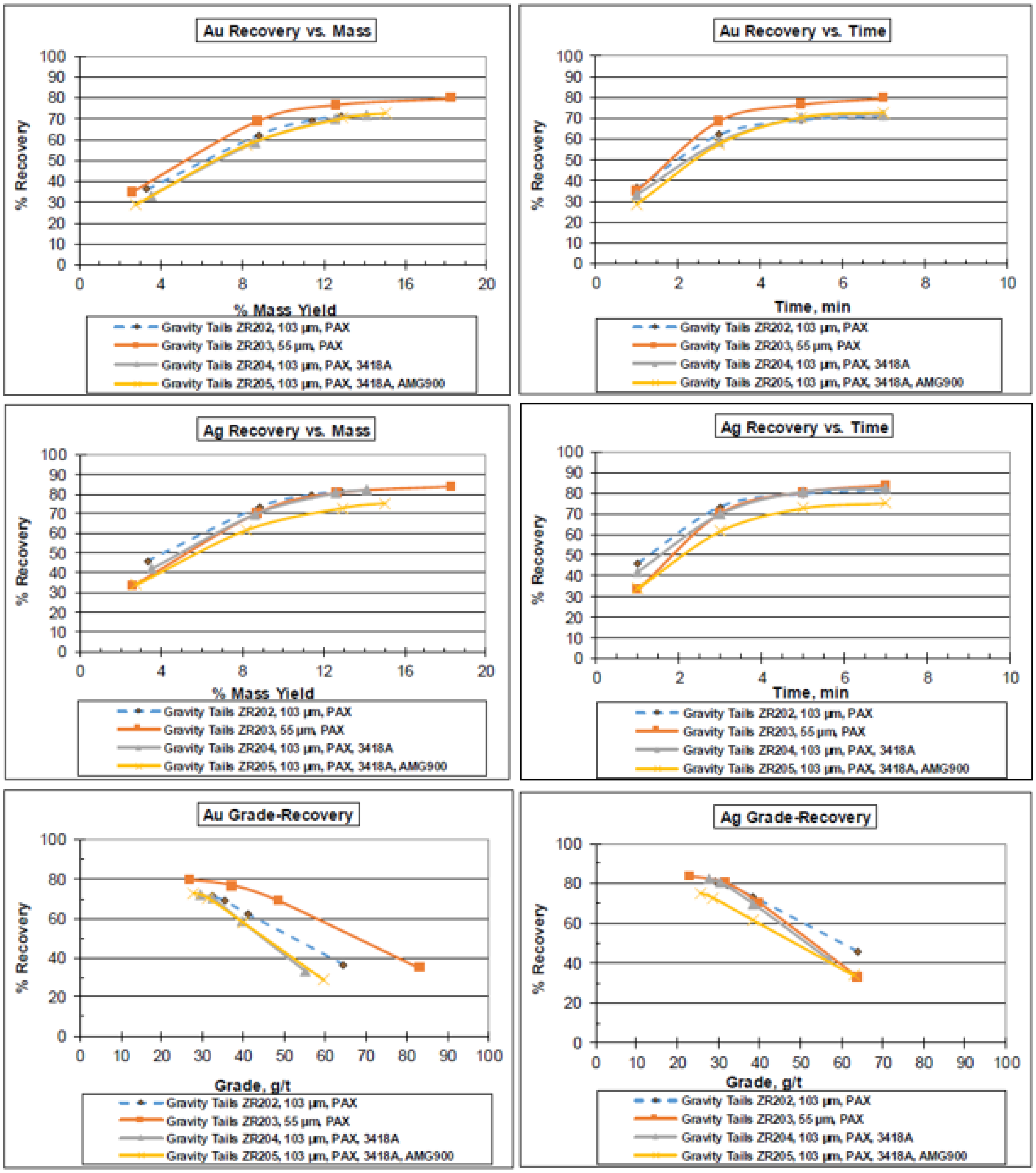


Figure 13-13: Rougher Flotation Kinetics and Grade-Recovery Relationship

The standard GRG test yielded a GRG value of 37.7% at a grind size of p80 103 µm. n der these conditions, the combined gravity and rougher flotation process achieved 82.2% gold recovery with a concentrate grade of 52.8 g/t Au in 14.3% mass, and 87.2% silver recovery with a grade of 40.8 g/t Ag.

When the grind size was reduced to p80 55 µm using the same reagent scheme, gold recovery improved to 87.2%, though the concentrate grade decreased to 42.1 g/t Au in 19.7% mass. Silver recovery also increased slightly to 88.6%, with a grade of 31.9 g/t Ag.

Overall, finer grinding of the gravity tailings enhanced gold recovery performance. However, the addition of reagents 3418A and AMG900 did not contribute to improved gold or silver recoveries.

Cleaner flotation tests were performed to evaluate the potential for producing a higher-grade gold concentrate suitable for downstream extraction. The first test (ZR212) followed the rougher conditions of Test ZR202 but with reduced mass pull during the rougher stage. Cleaner conditions were guided by visual observations to assess the upgradeability and cleaning behavior of the GRG tailings.

To enhance the cleaning efficiency observed in the first test, a second cleaner flotation test (ZR213) was conducted using regrinding of the rougher concentrate to a finer grind size of p80 55µm. The resulting grade-recovery relationships from both tests are presented in Figure 13-14.

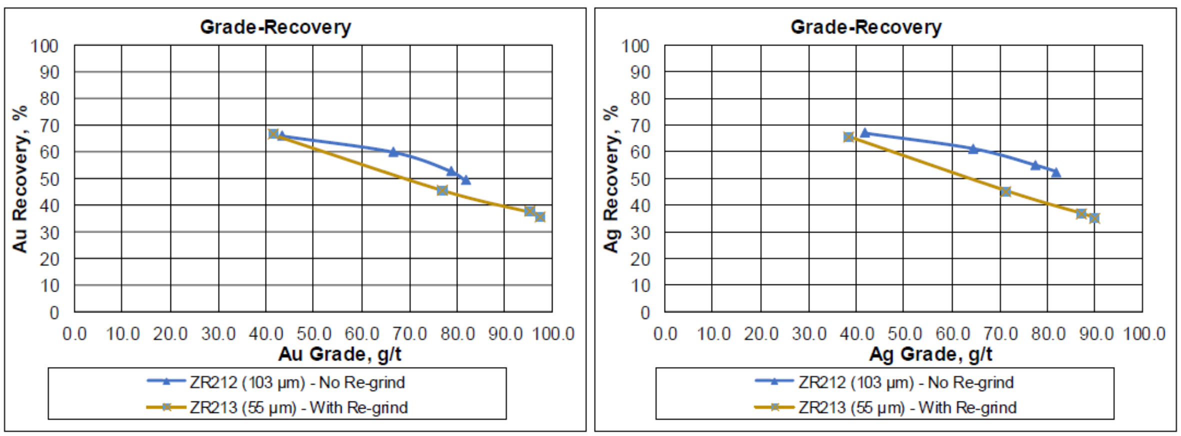


Figure 13-14: Cleaner Flotation Tests Grade-Recovery Relationship

The cleaner flotation tests demonstrated the impact of regrinding on gold concentrate grade and recovery. In the initial test (ZR212), conducted without regrinding, gold upgrading efficiencies across the first, second, and third cleaner stages were 1.53x, 1.81x, and 1.88x, respectively, resulting in a final gold recovery of 49.5% and a concentrate grade of 81.8 g/t Au.

In the second test (ZR213), regrinding the rougher concentrate to a p80 of 55µm improved upgrading efficiencies to 1.86x, 2.29x, and 2.35x, and increased the final concentrate grade to 97.5 g/t Au. However, this came at the cost of reduced gold recovery, which dropped to 35.7%, due to higher losses, specifically, a 21.0% loss to the 1st cleaner tailings stream.

The gravity-cleaner flotation process achieved a total gold recovery of 68.4% with a concentrate grade of 121.48 g/t Au in 5.2% mass during the third cleaner stage. Silver recovery reached 65.9%, with a grade of 96.5 g/t Ag.

When the rougher concentrate was re-ground to a finer size of p80 55 µm using the same reagent scheme, gold recovery decreased to 60.3%, although the concentrate grade improved to 145.2 g/t Au in 3.8% mass. Silver recovery also declined to 54.1%, with a higher grade of 106.5 g/t Ag.

Overall, finer re-grinding of the rougher concentrate did not benefit the gravity-cleaner flotation flowsheet, as it led to reduced recoveries of both gold and silver.

13.4.5.4 Gravity-Cyanide Leach Test Results

A 72-hour cyanide leach test (ZR302) was performed on the GRG tailings obtained from Test ZR102. Prior to cyanide addition, the tailings sample was re-pulped to 40% solids and conditioned with lime for one hour to stabilize the pH. This preparation ensured optimal conditions for the cyanidation process.

The leach test kinetics results are illustrated in Figure 13-15, demonstrated a stage gold recovery of 86.4% from the GRG tailings after leaching for 72 hours. This contributed an additional 56.8% gold recovery to the overall gold recovery by gravity-cyanidation treatment. The leach kinetics were fast, with 58.7% of the remaining gold extracted within the first three hours. Figure 13-15 shows the total recovery, which includes 34.3% gold and 31.1% silver recovery obtained from gravity concentration.

Extending the leaching time beyond 24 hours led to a slight increase in both gold and silver recovery. Additionally, reagent consumption was low, with sodium cyanide usage at 0.65 kg per tonne of solids and lime consumption at 0.72 kg per tonne of solids.

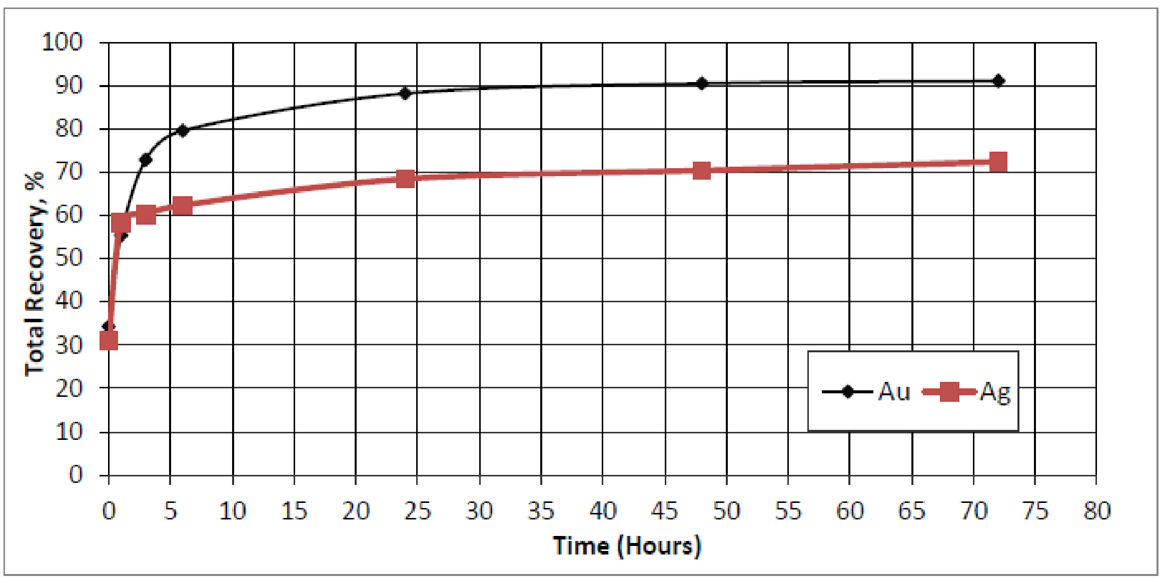


Figure 13-15: Cyanide Leach Kinetics on GRG Tailings

13.4.5.5 Whole Ore Leach Test Results

A 72-hour cyanide leach kinetics test (ZR303) was conducted on the head composite sample, ground to a fine particle size of 25 µm. This baseline test aimed to evaluate the maximum potential for gold and silver extraction through cyanide leaching. The results are illustrated in Figure 13-16, revealing that 97.6% of the gold was extracted over the 72-hour period.

The leaching process exhibited rapid kinetics initially, with 90.8% of the gold recovered within the first three hours. The rate of extraction slowed thereafter but continued steadily until the end of the test.

Compared to Test ZR302, which was performed on gravity tailings, Test ZR303 demonstrated significantly faster kinetics, largely attributed to the intensive grinding to 80% passing 25 microns.

Reagent consumption remained relatively low, with cyanide usage at 1.2 kg per tonne of feed and lime at 0.91 kg per tonne of feed. The test results suggest that some of the gold is associated with silicate minerals, which are not amenable to froth flotation. Given the ultra-fine grind size, it is probable that the residual gold in the final leach residue is finely disseminated and encapsulated within gangue minerals, limiting further recovery.

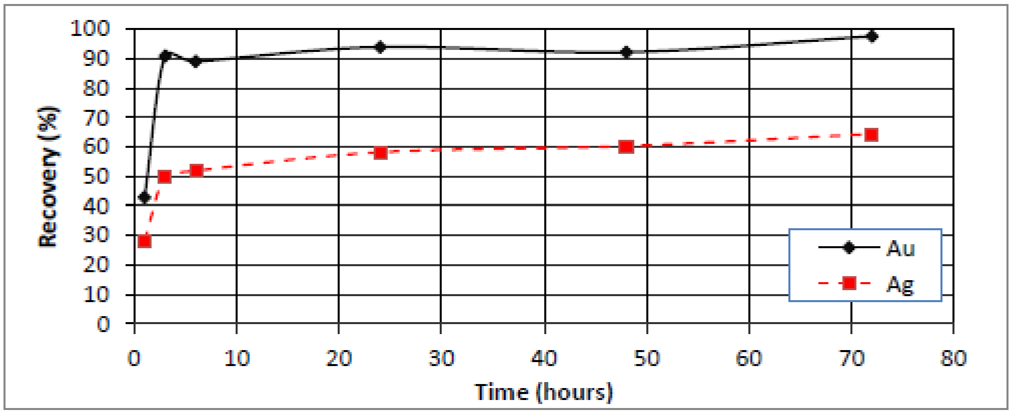


Figure 13-16: Whole Ore Cyanide Leach Kinetics

13.5 Historical Test Work (1981 to 1994)

13.5.1 1994 Mineralogical Examination of Scottie Gold Mine Samples

On March 31, 1994, Dr. Harlan Meade personally delivered three plastic bags containing mill-processed samples from the Scottie gold mine for mineralogical analysis. The primary goals of this examination were to assess the samples for any signs of oxidation-related alteration and to evaluate whether the pyrrhotite present exhibited any atypical compositional characteristics.

13.5.1.1 Samples

Of the three received sample bags, one was labeled "Westmin Resources Ltd. Premier Gold - 1991 Raw Ore Sample Scottie Gold - Pulverized," and another was marked "Westmin Resources Premier Gold - Scottie Tails." For the purposes of this report, these two are referred to as the Scottie ore and Scottie tails samples, respectively. The third bag contained cone-crushed material, which was excluded from the examination.

To ensure sample uniformity, each was thoroughly blended before approximately 100 grams were extracted for analysis. The selected portions were dried at room temperature on mylar sheets for three days, then repackaged into small plastic envelopes for submission for polished section preparation. The ore sample retained its pulverized texture throughout the drying process, while the tailings sample solidified into a mass that fractured into smaller pieces as it dried. These fragments were easily crushed by finger pressure, indicating that the cohesion was superficial and no substantial cementation had occurred.

Ultimately, one polished section was prepared from the head (ore) sample and one from the tailings sample for mineralogical analysis.

13.5.1.2 Results and Discussion

Both the ore and tailings samples exhibit similar mineralogical compositions, predominantly consisting of pyrite, quartz, and pyrrhotite, with smaller quantities of chlorite. Minor amounts of calcite are also present. Scanning electron microscopy revealed trace levels of arsenopyrite, sphalerite, chalcopyrite, apatite, galena, and rutile; however, these minerals are present in such low concentrations that they do not appear on the X-ray diffractograms of the bulk samples. The diffractograms display distinct peaks for calcite in both the head and tailings samples, while no peaks were observed for siderite ($FeCO_3$), dolomite ($CaMg(CO_3)_2$), or ankerite ($Ca(Fe,Mg)(CO_3)_2$). Therefore, CO_2 values obtained through chemical analysis can be reliably used to estimate calcite abundance for neutralization potential calculations.

Microscopic examination, including optical and scanning electron microscopy, showed that pyrrhotite grains possess sharp, unaltered margins. Electron microprobe analyses of pyrrhotite from both ore and tailings samples, revealed no detectable nickel or cobalt. Thirteen grains analyzed from the ore sample averaged 61.0% (w/w) iron and 38.8% (w/w) sulfur, totaling 99.8% (w/w), corresponding to the composition $Fe_{0.90}S_{1.0}$. Similarly, ten grains from the tailings averaged 60.9% (w/w) iron and 38.7% (w/w) sulfur, totaling 99.6% (w/w), also consistent with $Fe_{0.90}S_{1.0}$.

The Scottie pyrrhotite exhibits a consistent composition, averaging 47.37 atomic percent iron, placing it within the compositional range typical of hexagonal pyrrhotite. The samples analyzed appear pristine, showing no signs of oxidation-related alteration. Pyrrhotite is known to be significantly more prone to oxidation than pyrite, a trend supported by both field and laboratory studies.

In the Scottie tailings, the hexagonal form of pyrrhotite—though generally more stable than its monoclinic counterpart—still oxidizes at a rate that is tens to hundreds of times faster than pyrite. Consequently, during the early stages of oxidation, pyrrhotite will be affected first. Given the mineralogical makeup of the Scottie samples, any substantial oxidation should be easily detectable through visible "rusty" discoloration on the material's surface.

13.5.2 1992 Metallurgical Scoping Test work of Scottie Gold Mine Samples

Metallurgical testing has been carried out on an ore sample sourced from the underground muckpile at the Scottie Gold mine. The primary aim of this investigation was to assess whether the Scottie Gold ore is compatible with the existing operational procedures and equipment setup at the Premier Gold Mine.

13.5.2.1 Samples and Procedure

In early March 1991, an 80 kg sample of Scottie Gold ore was collected and transported to the Premier assay laboratory. Upon arrival, the sample was immediately dried in ovens to remove any visible moisture. It was then subjected to jaw and cone crushing, reducing the particle size to approximately 6 mm.

To facilitate handling and ensure sample uniformity, the crushed ore was coned, quartered, and riffled into 10 kg bags, each representing the overall composition of the original sample. From one of these bags, a portion was riffled and dry pulverized for a controlled duration. The pulverized material was subsequently screened at 53 microns to assess metallic gold. The resulting metal distributions and the back-calculated head grade are presented in Figure 13-17.

Table 13-17: Head Assay and Metal Distribution

Size (µm)	Sample (%)	Assay (g/t)		Distribution by Wt. (%)	
		Au	Ag	Au	Ag
+53	23.1	44.5	37.0	45.7	17.6
-53	76.9	15.9	52.0	54.3	82.4
Back Calculated	100.0	22.5	49.0	100.0	100.0

All leach tests followed a continuous monitoring protocol designed to track sodium cyanide (NaCN) and lime consumption throughout the duration of each test. Adjustments to cyanide and alkalinity levels were made at specific intervals 1, 3, 6, 24, and 48 hours—to maintain optimal leaching conditions. Additional samples were collected at the discretion of the testing personnel when deemed necessary.

This approach represents a standard method for continuously monitored leach testing. To replicate the current metal recovery conditions at the Premier Gold operation, fresh activated carbon was incorporated into all tests.

13.5.2.2 Grindability

Grindability testing, conducted in comparison with Premier's own open pit ore, shows that the Scottie Gold ore has a specific energy requirement of 10.9 kWh per metric tonne. This value is notably lower than the energy needed to grind Premier's ore, indicating that the Scottie material is easier to grind. The existing equipment and configuration at the Premier Milling Facility should be more than sufficient to handle the comminution of Scottie Gold ore effectively.

13.5.2.3 Cyanidation Results

A total of 42 leaching tests were conducted on an ore sample from the Scottie Gold deposit. These tests were designed to evaluate the compatibility of the Scottie ore with the operational setup of the Premier Gold Plant Facility. The primary objectives of the test program were to assess: (a) the leaching efficiency in relation to cyanide and lime consumption, (b) the impact of pre-aeration on leaching performance, and (c) the influence of lead nitrate addition during leaching.

The individual test condition and goals are presented in Table 13-18. The test result data are presented in Figure 13-17. Based on the collected data, it is evident that the gold in the Scottie ore is highly amenable to the leaching, with extraction reaching near completion within four hours. Silver, on the other hand, behaves similarly to the plant's native ore and requires extended leach times to achieve optimal recovery.

Table 13-18: Leach Test Conditions and Objectives

Test ID	Objective	Type
601	Binery Test, Does it leach? Y/N	CIL
602	Cyanide 500 g/t; To determine optimum cyanide addition	CIL
603	Cyanide 1000 g/t; To determine optimum cyanide addition	CIL
604	Cyanide 1500 g/t; To determine optimum cyanide addition	CIL
605	Cyanide 2000 g/t; To determine optimum cyanide addition	CIL
606	Cyanide 1500 g/t; with lead salt (150 g/t) to increase silver recovery	CIL
607	Cyanide 1500 g/t; with 6 hrs of pre-aeration to oxidize pulp and enhance recovery	CIL
608	Cyanide 1500 g/t; with lead salt (150 g/t); 6 hrs of pre-aeration	CIL
609	Cyanide 2000 g/t; with 6 hrs of pre-aeration to oxidize pulp and enhance recovery	CIL
610	Cyanide 100 g/t; To determine optimum cyanide addition	CIL
611	Cyanide 200 g/t; To determine optimum cyanide addition	CIL
612	Cyanide 300 g/t; To determine optimum cyanide addition	CIL
613	Cyanide 400 g/t; To determine optimum cyanide addition	CIL
614	Cyanide 500 g/t; To determine optimum cyanide addition	CIL
615	Cyanide 750 g/t; To determine optimum cyanide addition	CIL
616	Cyanide 1000 g/t; To determine optimum cyanide addition	CIL
617	Cyanide 250 g/t; To determine optimum cyanide addition, Repeat Test	CIL
618	Cyanide 500 g/t; To determine optimum cyanide addition, Repeat Test	CIL
619	Cyanide 750 g/t; To determine optimum cyanide addition, Repeat Test	CIL
620	Cyanide 1000 g/t; To determine optimum cyanide addition, Repeat Test	CIL
621	Cyanide 1250 g/t; To determine optimum cyanide addition, Repeat Test	CIL
622	Cyanide 1500 g/t; To determine optimum cyanide addition, Repeat Test	CIL
623	Cyanide 1750 g/t; To determine optimum cyanide addition, Repeat Test	CIL
624	Cyanide 2000 g/t; To determine optimum cyanide addition, Repeat Test	CIL
625	Cyanide 2250 g/t; 2 hrs leaching	CIL
626	Cyanide 2250 g/t; 4 hrs leaching	CIL
627	Cyanide 2250 g/t; 6 hrs leaching	CIL
628	Cyanide 2250 g/t; 24 hrs leaching	CIL
629	Cyanide 2250 g/t; 48 hrs leaching	CIL

Test ID	Objective	Type
630	Cyanide 2250 g/t; Preg-robbing, 8 hrs leaching, 40 hrs CIL	CIP
631	Cyanide 2250 g/t; Preg-robbing, 48 hrs leaching CIL (Background Test for 630)	CIL
632	Cyanide 3000 g/t; To determine optimum cyanide addition, Repeat Test	CIL
641	Cyanide 1500 g/t; To determine optimum cyanide addition with pH Control CaO at 2000 g/t	CIL
642	Cyanide 1500 g/t; To determine optimum cyanide addition with pH Control CaO at 2500 g/t	CIL
643	Cyanide 1500 g/t; To determine optimum cyanide addition with pH Control CaO at 2000 g/t	CIL
644	Cyanide 1500 g/t; To determine optimum cyanide addition with pH Control CaO at 2500 g/t	CIL
645	Cyanide 1500 g/t; To determine optimum cyanide addition with Pre-Aeration at pH Control CaO at 2000 g/t	CIL
646	Cyanide 2000 g/t; CaO at 2500 g/t; Repeat Test 644	CIL
647	Cyanide 2000 g/t; CaO at 2500 g/t; Repeat Test 644	CIL
650	Cyanide 2500 g/t; Pulp first acidified at pH 2 for 20 hrs then neutralized to pH 11.6 before cyanidation	CIL
651	Cyanide 2500 g/t; Pulp first pre-aerated for 9 hrs at natural pH 7 before cyanidation	CIL
652	Cyanide 2500 g/t; Pulp first pre-aerated for 9 hrs at elevated pH 11.6 before cyanidation	CIL

Pre-aeration showed minimal improvement in metal extraction rates (as observed in Tests 607, 609, 645, 651, and 652), but it did contribute to reduced cyanide consumption. The addition of lead nitrate, intended to bind sulphates and enhance silver recovery, had little to no beneficial effect (Tests 606 and 608). In fact, Test 608 indicated that introducing lead nitrate after pre-aeration may have hindered silver leaching, likely due to the reprecipitation of soluble silver by lead.

To optimize the recovery of both gold and silver from the Scottie Gold ore, a cyanide dosage of 1500 g NaCN per metric tonne of feed and a lime dosage of 1500 g per metric tonne of feed are recommended.

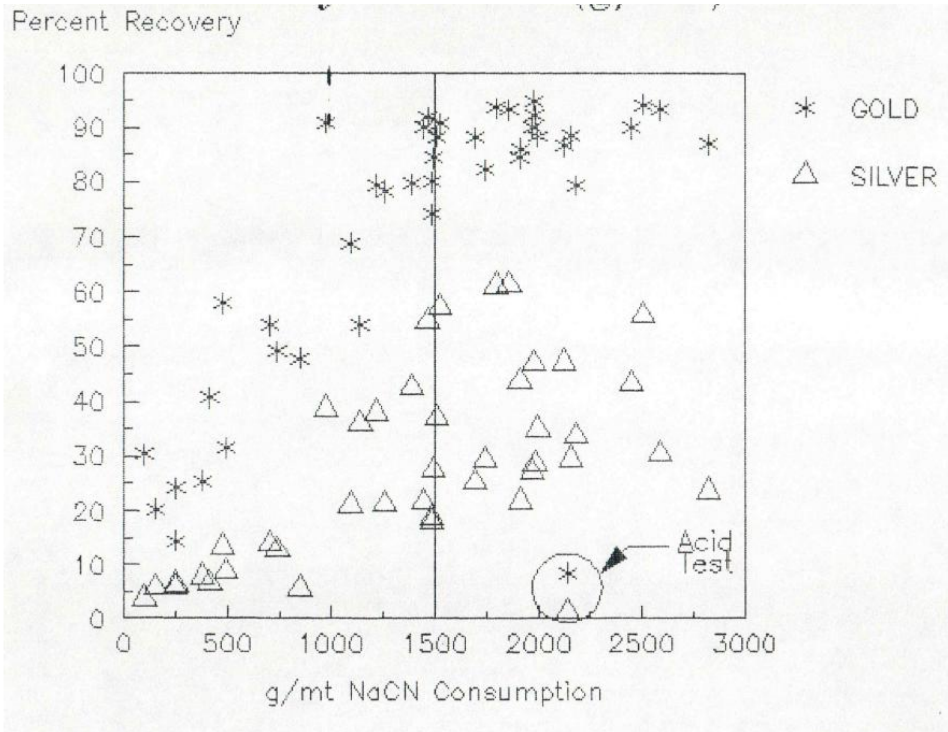


Figure 13-17: Leach Test Results

13.5.2.4 Cyanide Destruction

Scottie Gold represents one of the earliest successful implementations of the INCO SO_2/air process for cyanide destruction. Laboratory batch testing of cyanide destruction confirms that the material should be amenable to the plant cyanide destruction system.

According to the leach solution metal analysis, the data suggested that copper sulfate ($CuSO_4$) consumption will likely exceed the addition levels at Premier due to elevated concentrations of ferricyanide ($Fe(CN)_6$) in the leach tailings. Sulfur dioxide (SO_2) consumption is projected to range between 6.0 and 10.0 grams of SO_2 per gram of cyanide treated.

Maintaining optimal control over slurry alkalinity is critical, as it can significantly reduce both cyanide usage and the associated costs of cyanide destruction.

13.5.2.5 Solids Settling

Settling tests confirmed that the Scottie Gold ore was compatible with the flocculant utilized at the Premier facility at that time. Column settling test data is depicted in Figure 13-18, supports this conclusion. Based on the results, the existing flocculant dosage of 60 grams per tonne—used for Premier ore—should remain unchanged.

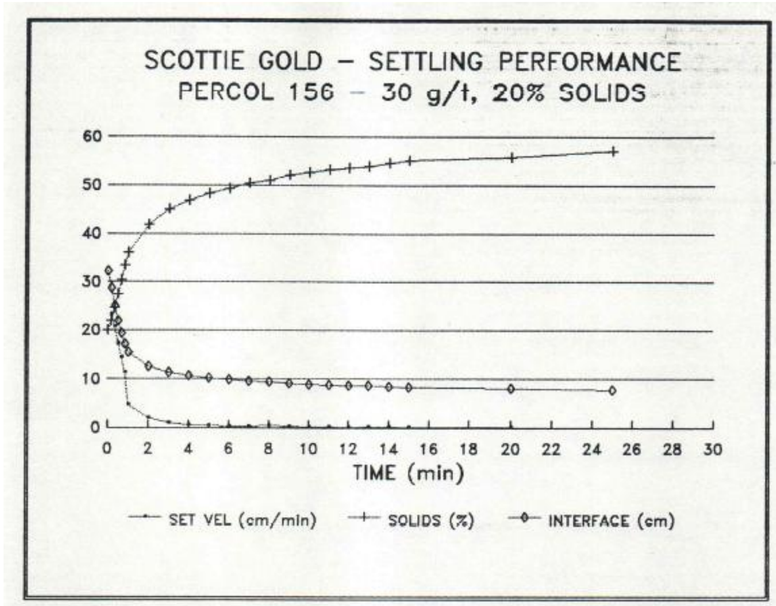


Figure 13-18: Settling Performance of Scottie Gold Sample

13.5.2.6 Environmental

Environmental assessments indicate that the Scottie Gold ore has a high potential to generate acid mine drainage, based on Acid/Base Accounting test results. The stability of the Scottie Gold tailings within the Premier tailings pond would largely depend on the method and conditions under which the pond is ultimately decommissioned. Proper planning and management of the decommissioning process was expected to be essential to mitigate potential acid generation and ensure environmental compliance.

13.5.3 1981 Mineralographic Study of Massive Sulphide Chips of Scottie Gold Mine Samples

Three small chips of regressive sulphide material were submitted by Scottie Gold Mines Ltd. for mineralogical examination, with the primary focus on identifying and characterizing the gold and silver mineralogy. The submitted samples included both pyrite-rich and pyrite-poor material. To facilitate detailed analysis, a polished surface was prepared for each type, allowing for comparative study of their mineralogical features.

13.5.3.1 Mineralogy Result

Specimen A is sphalerite-rich and composed primarily of massive sulphides, with sphalerite as the dominant mineral phase. It contained approximately 25% pyrite and 15% galena, and its surface showed rusty weathering, suggesting oxidation. The grain size is medium throughout. Specimen B, in contrast, was pyrite-rich and also consisted of massive sulphides, with pyrite as the principal component and 5–10% sphalerite. It shared the same medium-grained texture as Specimen A.

Native gold was definitively identified in four grains within sphalerite in one specimen and tentatively observed as an extremely small grain in sphalerite in a second specimen. The confirmed grains measured 0.10 × 0.04 mm², 0.03 × 0.05 mm², 0.02 × 0.01 mm², and less than 0.01 × 0.01 mm². The occurrence of gold does not appear to be controlled by fractures in this mineral association. Notably, one of the gold grains was found in association with a single grain of argentite.

Remaining gangue minerals constituted a 5 percent of the overall composition, and the results are listed in Table 13-19.

Textural relationships in the examined specimens do not clearly reveal the paragenetic sequence of mineral formation. However, pyrite, arsenopyrite, sphalerite, and pyrrhotite appear to form an early-stage sulphide assemblage, potentially associated with gold mineralization. In contrast, galena and tetrahedrite seem to represent a later-stage sulphide association. Silver may be linked to both assemblages, given the known association of argentite with gold and tetrahedrite to galena.

Table 13-19: Polished Surface with Visual Modal Analysis

Mineral	Specimen A (%)	Specimen B (%)
Pyrite	15	67
Sphalerite	63	15
Arsenopyrite	5	3
Galena	10	10
Pyrrhotite	1	Trace
Chalcopyrite	0.5	Trace
Gold	Trace	Trace
Tetrahedrite	Trace	Trace
Argentite	Trace	Trace

14.0 MINERAL RESOURCE ESTIMATES

The Mineral Resource Estimate (MRE) for the Scottie Gold Mine Project has an effective date of February 2, 2025. The resource estimate was prepared by Sue Bird, P.Eng., of Moose Mountain Technical Services (MMTS).

14.1 Mineral Resource Estimate

The Scottie Project total MRE includes the Scottie and Blueberry deposits, with the Blueberry containing satellite deposits called the Bend and Gulley zones. The MRE is summarized in Table 14-1 for the base case cut-off grade. Mineral Resources were estimated using the 2019 CIM Best Practice Guidelines and are reported using the 2014 CIM Definition Standards.

The resource utilizes pit shells to constrain resources at the Blueberry deposits and potentially minable underground shapes at varying cutoff grades to define the underground resource below the Blueberry pit and for the Scottie Mine underground resources. The current estimate uses metal prices of US$2,000/oz gold price, recoveries, smelter terms and costs, as summarized in the notes to the resource table. Metal prices have been chosen based partially on three-year trailing averages and industry standard pricing currently used for resource estimates. The Au price chosen also considered the spot prices and the three-year trailing average prices.

The base case cut-off grade for open pit mining is 0.70 g/t Au and 2.5 g/t Au for underground resources, which more than covers the Processing + G&A for the open pit mining and covers costs of Processing + G&A + underground development costs for the underground resource.

These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The QP is of the opinion that issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work. These factors may include environmental permitting, infrastructure, sociopolitical, marketing, or other relevant factors.

As a point of reference, the in-situ gold are inventoried and reported by intended processing method.

Figure 14-1 illustrate a three-dimensional view of the block model showing the Au grade above cutoff and the resource open pit and underground resource shapes for the blueberry, Bend and Gulley deposits. Figure 14-2 illustrates the blocks above cutoff and underground resource shapes for the Scottie Mine deposit, as well as the previous underground workings and stopes.

The resource estimate for the PEA is based on inferred resources as stated in the February 2025 Resource Estimate for the Scottie Gold Mine project. Certain mining factors have been applied to this resource estimate, to generate diluted resources using a conceptual mine plan for the PEA. The February 2025 Resource Estimate is summarized below:

Table 14-1: Mineral Resource Estimate for the Scottie Gold Mine Project

Blueberry Pit Resource					
Source	**Cutoff Au (g/t)**	**Tonnage (ktonnes)**	**Au (g/t)**	**NSR ($CDN)**	**Au Metal (kOz)**
Blueberry Pit (Inferred)	0.25	2,887	2.06	156.04	191
	0.3	2,712	2.17	164.69	190
	0.5	2,114	2.68	202.51	182
	0.7	**1,707**	**3.17**	**239.73**	**174**
	1	1,323	3.85	290.19	164
	2.5	600	6.61	492.83	128
	5	273	10.35	755	91
Total Underground Resource					
Source	**Cutoff Au (g/t)**	**Tonnage (ktonnes)**	**Au (g/t)**	**NSR ($CDN)**	**Au Metal (kOz)**
Blueberry and Scottie Mine Underground (Inferred)	**2.5**	**1,897**	**8.66**	**678.51**	**528**
	3	1,704	9.33	731	511
	3.5	1,549	9.94	778.78	495
	4	1,404	10.59	829.04	478
	4.5	1,269	11.26	881.69	459
	5	1,143	11.98	937.99	440
	10	520	18.05	1,413.75	302
Total	**varies**	**3,604**	**6.06**	**470.69**	**703**

Notes to the 2025 Resource Table:

1. Resources are reported using the 2014 CIM Definition Standards and were estimated using the 2019 CIM Best Practices Guidelines, as required National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”)
2. The base case MRE has been confined by “reasonable prospects of eventual economic extraction” shape using the following assumptions:
    - Metal price of US$2000/oz gold
    - Metallurgical recovery of 90% gold
    - Payable metal of 99% gold in doré
    - Forex of 0.74 $US:$CDN
    - Processing costs of CDN$24 / tonne milled, which includes milling, transport, smelter treatment, refining and General & Administrative (G&A) costs
    - Underground production cost of CDN$78 / tonne, and underground development costs to be CDN$90 / tonne, for a total underground mining cost of CDN$168 / tonne
    - Open pit mining costs of CDN$3.00 / tonne for mineralized and waste material

- 45-degree pit slopes
- The 130% price case pit shell is used for the confining shape with elevation adjustment of the main Blueberry pit for the underground resource.

3. The resulting net smelter return is NSR = Au g/t* CDN$98.60 / g * 90% recovery rate
4. Numbers may not add due to rounding
5. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the estimated mineral resources will be converted into mineral reserves.

It is noted that the prices and costs used for the resource estimate are not identical to the updated values used for the mining and cash flow calculations. The costs and Au price used for the resource reflect those considered reasonable at the effective date of the resource estimate. A check has been done on these values compared to the current values and it is found that the resource is somewhat conservative on price resulting in a similar cutoff for open pit mining and somewhat higher cutoff for underground than that used for the mining study and cashflow.

The Qualified Person is of the opinion that issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work. These factors may include environmental permitting, infrastructure, sociopolitical, marketing, or other relevant factors.

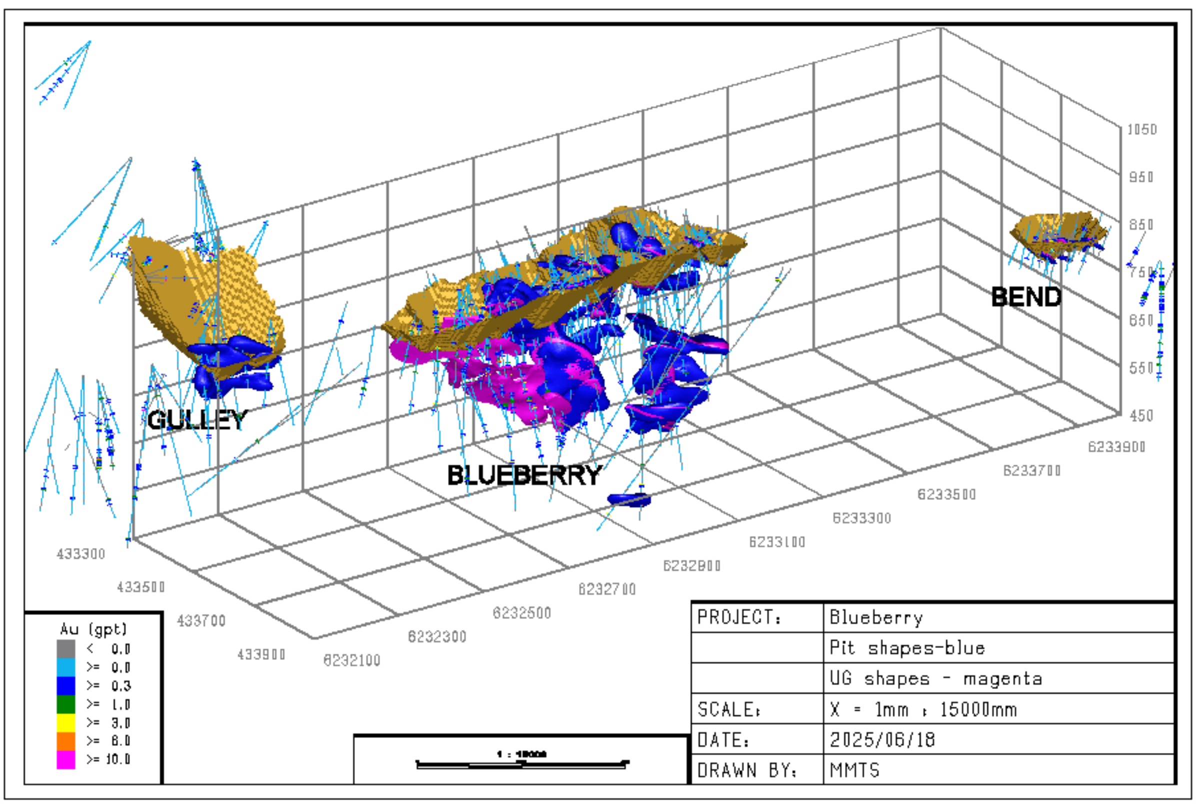


Figure 14-1: Blueberry/Bend/Gulley Resource Pit and Modelled Au Grades

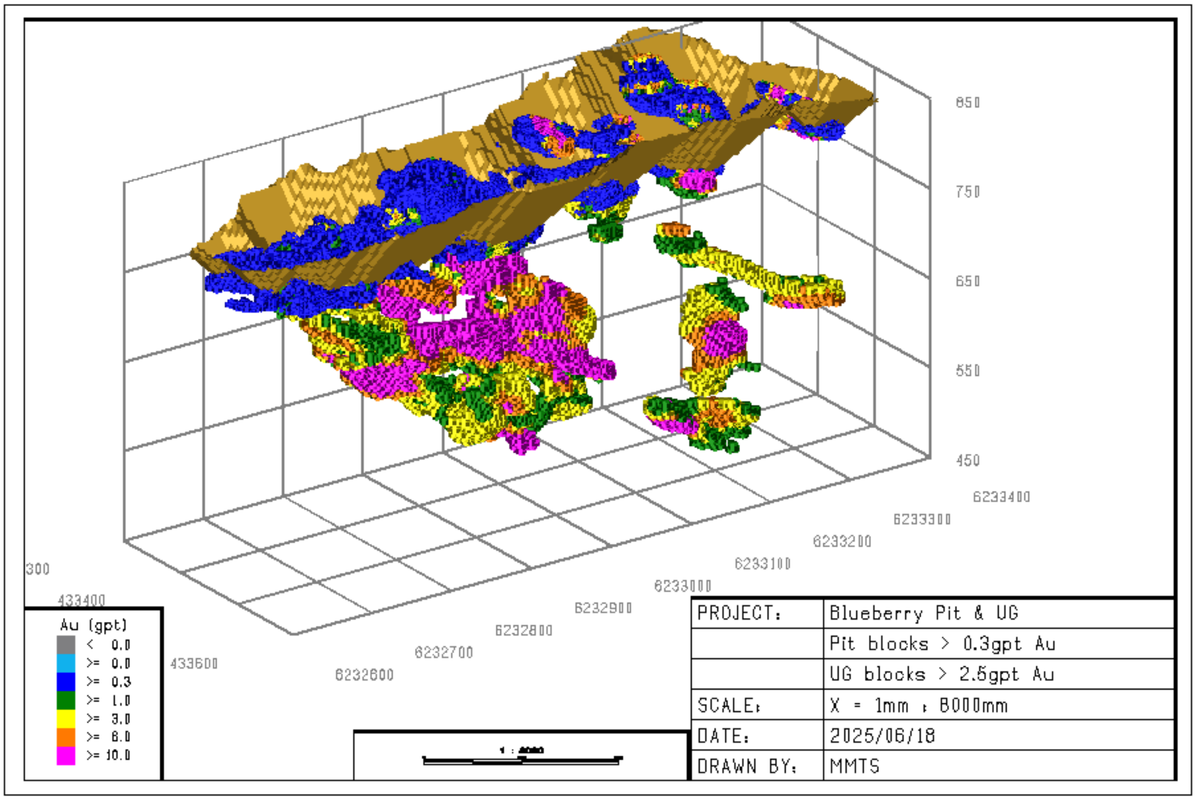


Figure 14-2: Blueberry Open Pit and Underground

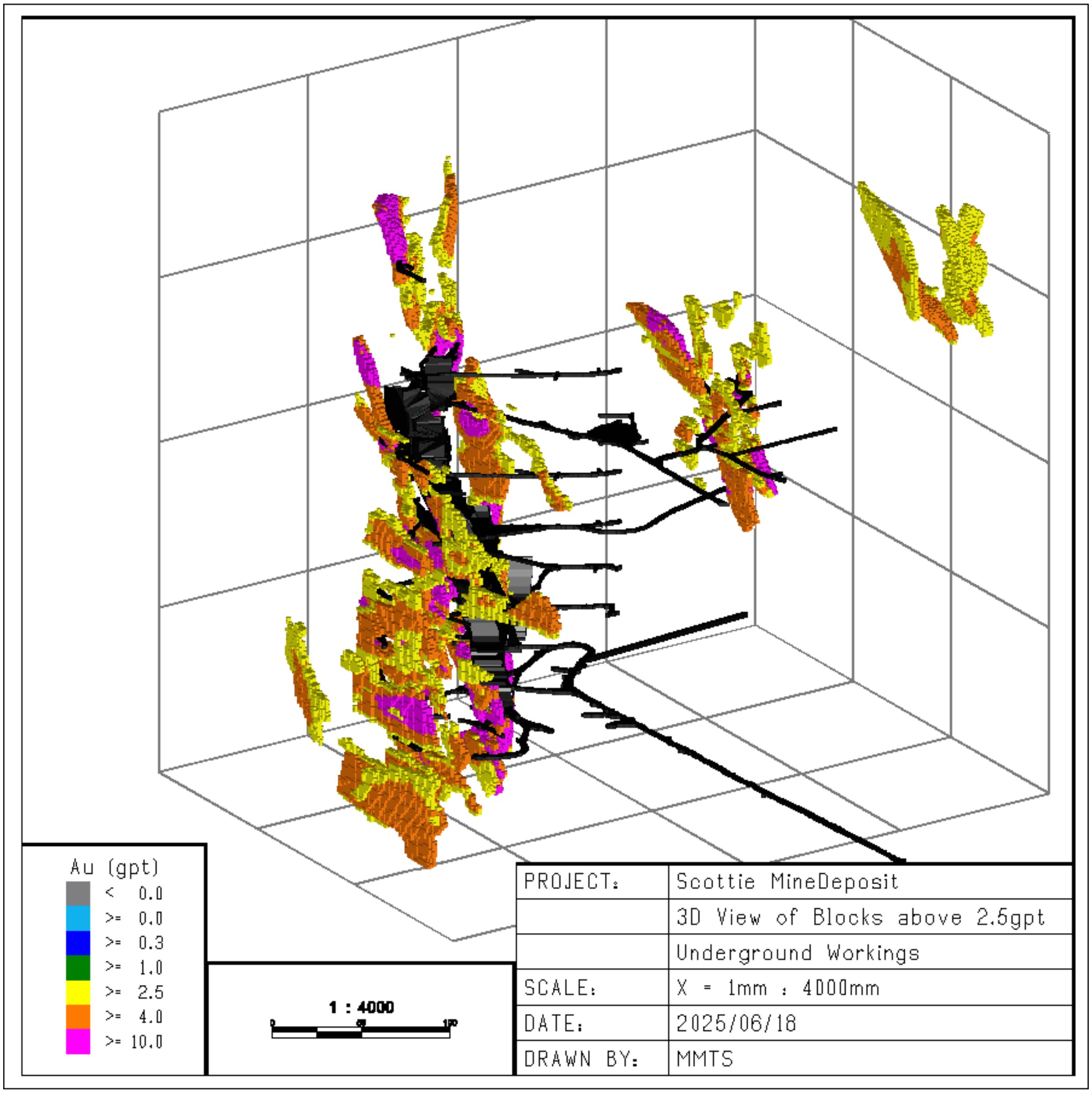


Figure 14-3: Scottie Deposit Underground Resource

14.2 Key Assumptions and Data Used in the Estimate

The total Blueberry and Scottie Mine modelled areas comprises a database as summarized in Table 14-2. Drilling since 2016 has been done by Scottie Resources and is summarized in Section 10. Previous historical drilling is described in Section 6 and has been validated as summarized in Section 12.

Table 14-2: Summary of Scottie Project Drillhole within Block Models

Area	Year	# DDH	Length (m)	Within the Database		Within the Domains	
				# Assays	Total Assay Length (m)	# Assays	Total Assay Length (m)
Blueberry/ Bend/ Gulley	Undefined	8	535.5	0	0.0	0	0.0
	1983	8	592.8	1,381	877.8	0	0.0
	1984	18	1,018.3	1,458	938.6	77	59.8
	2005	16	639.7	1,237	1,227.0	113	94.9
	2006	3	207.6	539	572.9	67	56.1
	2016	18	1,940.0	663	804.6	0	0.0
	2019	18	1,494.4	890	1,490.1	97	116.6
	2020	15	1,939.7	4,368	4,149.8	320	286.2
	2021	48	5,652.3	9,970	9,637.6	1,142	1,013.1
	2022	86	16,562.9	11,946	12,119.4	2,017	1,986.4
	2023	81	19,519.2	12,544	15,131.4	1,666	1,736.9
	2024	24	5,875.0	7,725	8,881.7	457	493.9
	Total	**343**	**55,977.4**	**52,721**	**55,830.7**	**5,956**	**5,843.7**
Scottie	Undefined	21	1,482.8	0	0.0	0	0.0
	1948	19	1,627.6	62	59.0	29	26.4
	1979	22	932.1	226	135.2	71	39.3
	1981	28	588.2	123	139.3	24	24.8
	1982	67	4,139.2	719	615.6	173	141.1
	1983	60	6,293.2	1,171	736.9	273	162.0
	1984	99	7,150.7	1,250	745.7	365	232.1
	1987	20	1,975.3	263	194.9	47	33.6
	2004	14	1,273.8	501	505.7	40	31.5
	2005	29	3,169.6	1,019	1,029.6	102	73.6
	2006	13	2,365.9	446	491.3	14	10.4
	2019	1	539.1	235	501.0	11	11.0
	2020	10	2,815.0	903	987.9	66	53.4
	2021	17	5,179.7	2,818	3,744.6	78	82.5
	2022	2	386.4	241	334.7	0	0.0
	2023	3	648.4	338	638.5	4	5.7
	2024	20	4,395.0	2,059	3,878.4	140	183.1
	Total	**445**	**44,962.0**	**12,374**	**14,738.2**	**1,437**	**1,110.4**

14.3 Geological Modelling

Three-dimensional wireframe solids based on geology have been used to constrain the grade interpolations. At the Scottie and Blueberry deposits three-dimensional solids mineralization domains have been created to confine the resource. Figure 14-3 illustrates the Au mineralization domains for the Scottie mine deposit. For the Blueberry deposit, two sets of domains have been created, lower grade Au domains (shown in blue) to confine the open pit resource estimate near surface, and higher-grade Au domains (shown in magenta) for the potential underground resource at depth as illustrated in Figure 14-4.

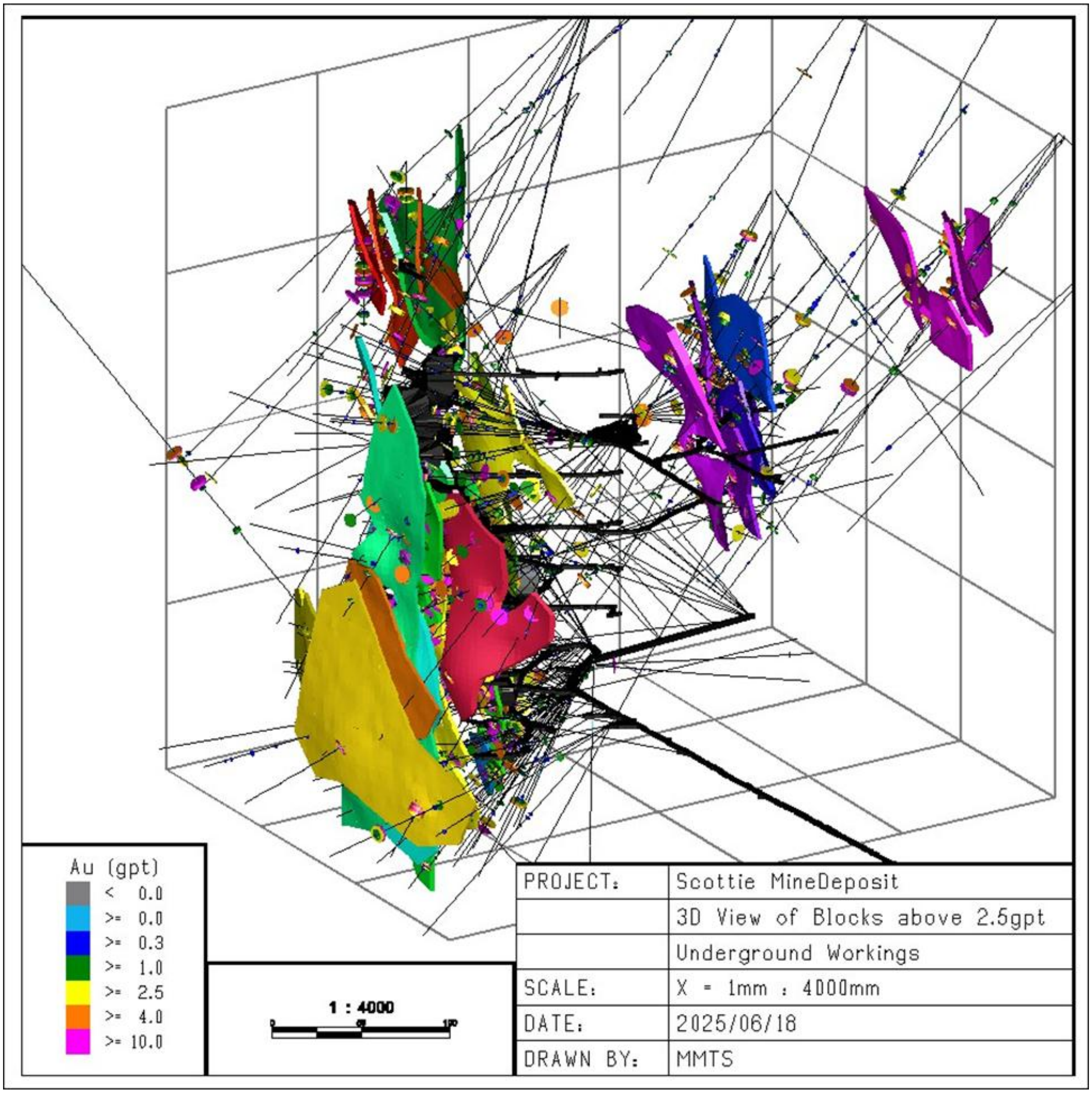


Figure 14-4: Scottie Mine Deposit – Au Domains

(Source: MMTS, 2025)

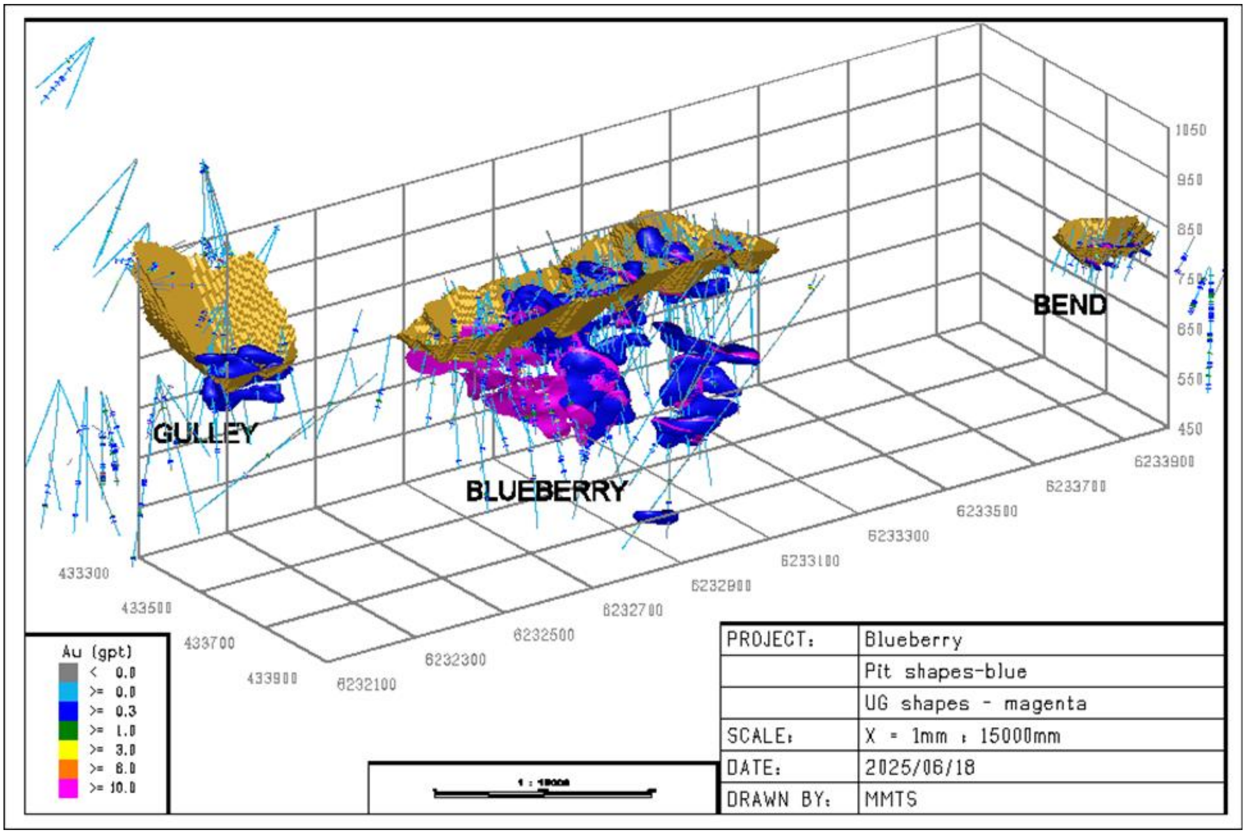


Figure 14-5: Blueberry Area Deposits – Open Pit and Underground Mineralization Domains
(Source: MMTS, 2025)

14.4 Capping

Cumulative probability plots (CPP) are used to define capping values and potential outlier restrictions during interpolations. Figure 14-5 and show in the examples of the CPP plots for Au for selected domains with the Scottie mine deposit. Figure 14-6 shows examples of the CPP plots for Au in selected Blueberry area domains, with the higher-grade domains modelled for the underground resource in reds and the lower grade domains modelled for the open pit resource in blues.

Capping and Outlier values are summarized in Table 14-3 for Blueberry domains and in Table 14-4 for the Scottie domains. Values above the capping value are equal to the capped value in the assay file prior to compositing. Composite values above the Outlier value are restricted during interpolations to the Outlier value for distance greater than 10 m from the composite interval.

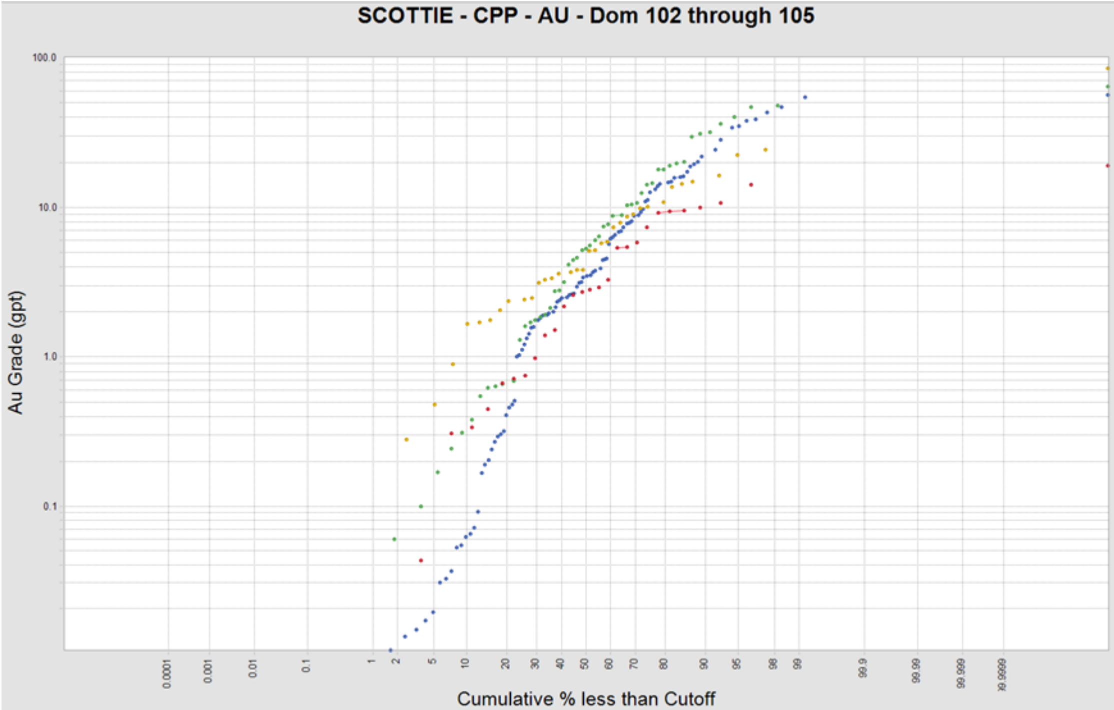


Figure 14-6: CPP Examples of Au Assay Data by Domain – Scottie Deposit

(Source: MMTS, 2024)

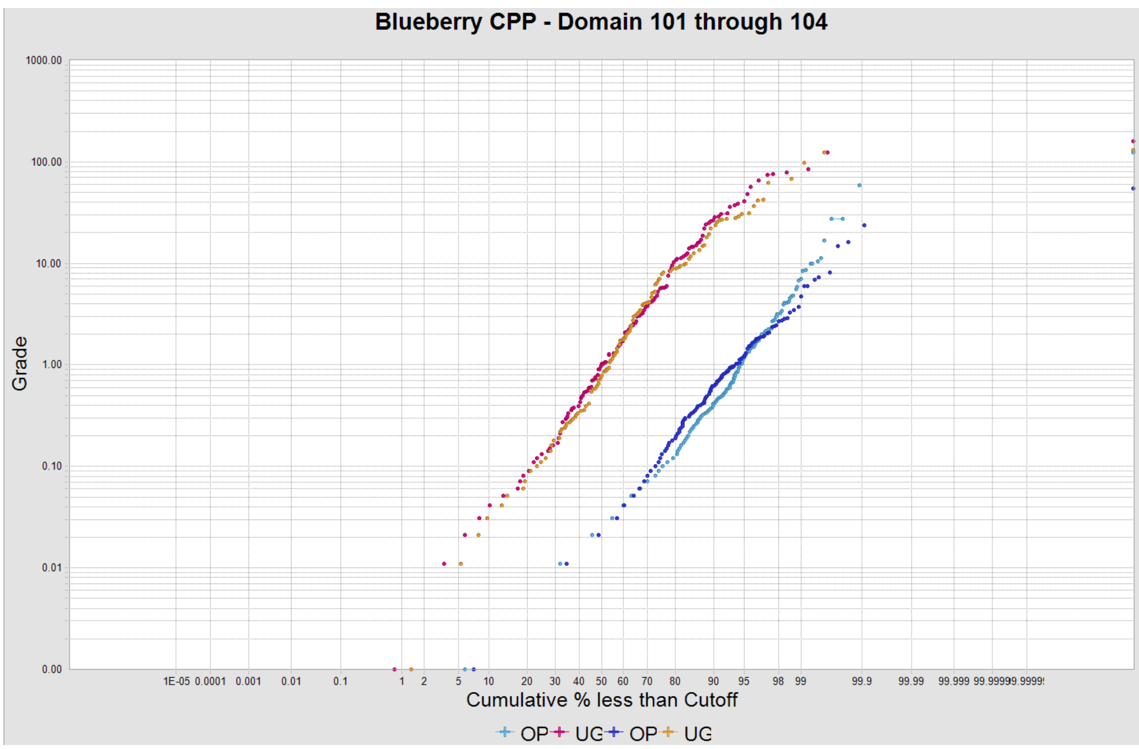


Figure 14-7: CPP Examples of Au Assay Data by Domain – Blueberry Open Pit Domains

(Source, MMTS, 2024)

Table 14-3: Summary of Capping and Outlier Restriction Values – Blueberry

Low Grade Halo Domain			High Grade Core Domain		
Domain	Au Cap Grade (g/t)	Au Outlier Grade (g/t)	Domain	Au Cap Grade (g/t)	Au Outlier Grade (g/t)
101	60	9	102	80	–
103	30	7	104	100	35
105	30	–	106	30	–
107	6	–	108	30	–
109	30	–	110	80	–
111	0.7	–	112	30	–
113	20	–	114	100	–
115	2	–	116	50	–
117	10	–	118	40	–
119	30	–	120	NA	–
121	6	–	122	NA	–
123	2	–	124	30	–
125	2	–	126	NA	–
127	20	–	128	NA	–
129	5	–	130	NA	–
131	7	–	132	NA	–
133	7	–	134	NA	–
135	5	–	136	NA	–
137	7	–	138	NA	–
139	5	–	140	30	–
141	3	0.9	142	30	13
143	50	–	144	110	–
145	5	–			

Table 14-4: Summary of Capping and Outlier Restriction Values – Scottie

Domain	Au Cap Grade (g/t)	Au Outlier Grade (g/t)	Domain	Au Cap Grade (g/t)	Au Outlier Grade (g/t)
1	12	80	107	30	80
2	10	80	108	150	80
3	22	80	109	20	80
4	100	80	110	100	80
5	30	80	111	60	80
6	50	80	112	20	80
7	60	80	113	10	80
8	10	80	114	10	80
101	20	80	115	100	80
102	100	80	117	100	80
103	100	80	118	10	80
104	40	80	119	10	80
105	50	80	120	4	80
106	100	80	121	2	80

The capped assay and composite statistics of each domain are summarized in Table 14-5 for both Blueberry model and the Scottie model drillhole data. The table illustrates that no significant bias has been introduced during the compositing process.

Table 14-5: Capped Assay and Composite Statistics

Parameter	Scottie		Blueberry	
	Assays	Comps	Assays	Comps
Num Samples	7,422	902	6,353	3,235
Num Missing Samples	0	0	0	0
Min	0	0	0	0
Max	150.0	117.8	110.0	78.0
Weighted mean	5.594	5.594	1.422	1.422
Difference (%)	**0.0%**		**0.0%**	

14.5 Compositing

Compositing of Au grades has been done as 2.0 m fixed length composites. Small intervals less than 1.0 m are merged with the up-hole composite if the composite length is less than 1.0 m. The length of 1.0 m is chosen to be half the size of the block height, and longer than the majority of assay lengths, as illustrated in Figure 14-7. Domain boundaries are honored during compositing.

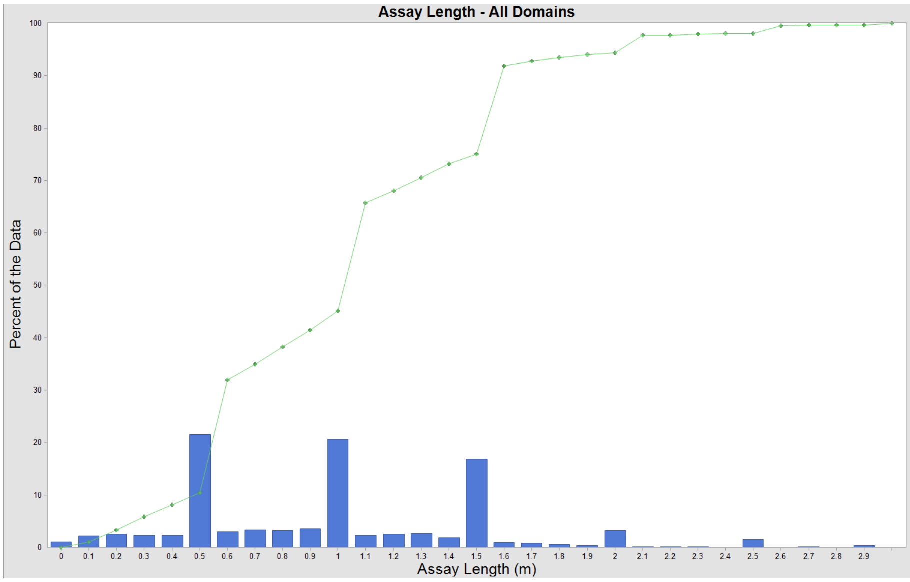


Figure 14-8: Histogram of Assay Lengths – Scottie Project Domains
(Source: MMTS, 2025)

14.6 Block Model Interpolations

The block model limits and block size for each deposit are as given in Table 14-6.

Table 14-6: Block Model Limits

Deposit Area	Direction	Minimum	Maximum	Size	# Blocks
Scottie	Easting	431,700	432,100	2	200
	Northing	6,230,950	6,231,400	2	225
	Elevation	770	1,200	2	215
Blueberry	Easting	432,400	434,100	5	340
	Northing	6,231,750	6,234,250	5	500
	Elevation	200	1,250	5	210

Interpolation of Au is done by inverse distance squared (ID2) in four passes based on the orientations of the modelled veins and mineralization zones. Interpolations used hard boundaries, with composites and block codes required to match within each domain. Search parameters are summarized in Table 14-7 through Table 14-9 below.

Table 14-7: Search Rotations by Domain – Blueberry

Low Grade Halo Domain				High Grade Core Domain			
Domain	Y	X	Z	Domain	Y	X	Z
101	350	-45	0	102	350	-45	0
103	340	-50	0	104	340	-50	0
105	320	-40	0	106	320	-40	0
107	320	-40	0	108	320	-40	0
109	315	-40	0	110	315	-40	0

Low Grade Halo Domain				High Grade Core Domain			
Domain	Y	X	Z	Domain	Y	X	Z
111	290	-55	0	114	290	-35	0
113	290	-35	0	116	325	-40	0
115	325	-40	0	118	330	-35	0
117	330	-35	0	124	300	-55	0
119	310	-35	0	140	340	-40	0
121	280	-65	0	142	350	-40	0
123	300	-55	0	144	330	-40	0
125	290	-65	0	102	350	-45	0
127	305	-55	0	104	340	-50	0
129	305	-35	0	106	320	-40	0
131	245	-40	0	108	320	-40	0
133	290	-50	0	110	315	-40	0
135	280	-40	0	114	290	-35	0
137	0	-40	0	116	325	-40	0
139	340	-40	0	118	330	-35	0
141	350	-40	0	124	300	-55	0
143	330	-40	0	140	340	-40	0
145	335	-50	0	142	350	-40	0
				144	330	-40	0

Table 14-8: Search Rotation and Distances – Scottie

Domain	Y	X	Z	Domain	Y	X	Z
1	285	0	-56	107	320	0	85
2	300	0	-55	108	307	0	-85
3	280	0	-70	109	320	0	85
4	280	0	-70	110	290	0	-70
5	280	0	-70	111	277	0	-80
6	280	0	-70	112	290	0	-70
7	280	0	-70	113	290	0	-70
8	280	0	-70	114	265	0	-80
101	290	0	-70	115	294	0	-70
102	310	0	-75	117	266	0	-85
103	265	0	-80	118	265	0	-80
104	265	0	-80	119	307	0	-85
105	265	0	-80	120	307	0	-85
106	290	0	-70	121	294	0	-70

Table 14-9: Additional Search Criteria

Criteria	Pass 1	Pass 2	Pass 3	Pass 4
Distance - Y	10	20	50	100
Distance - X	5	10	25	50
Distance - Z	5	10	25	50
Minimum # composites	3	3	3	3
Maximum # Composites	12	12	12	12
Maximum / drillhole	2	2	2	2
Maximum / quadrant	2	2	2	NA

14.7 Classification

Classification of both deposits is considered to be all Inferred.

14.8 Block Model Validations

14.8.1 Comparison to De-clustered Composites

Interpolations have also been completed using a Nearest Neighbour method to essentially de-cluster the composite data for grade comparisons with the modelled grades. Table 14-10 gives a summary of the mean grades for de-clustered composites (NN interpolation), and OK grades. The modelled grades are conservative when compared to the un-capped Nearest neighbour grades, as is considered appropriate for nuggety high grade gold deposits.

This comparison is illustrated more succinctly in the swath plots. Mean grades across the model in both Northing and Easting directions are calculated and compared in Figure 14-8 through Figure 14-11 for both the Blueberry deposits and Scottie Mine.

Table 14-10: Comparison of De-clustered Composite and OK Modelled Grades

Parameter	Scottie		Blueberry			
	AU	AUNN	AU1-LG	AUNN1-LG	AU2-HG	AUNN2-HG
Num Samples	116,967	120,686	79,206	78,866	17,938	16,635
Num Missing	30,852	27,133	1,388	1,728	1,299	2,602
Min (g/t)	0.000	0.010	0.000	0.000	0.010	0.002
Max (g/t)	105.993	258.697	50.101	68.710	68.425	81.238
Wtd mean (g/t)	4.342	8.431	1.122	1.732	6.762	7.462
Weighted CV	1.33	1.83	1.99	2.91	1.02	1.03
Difference (%)	**-49%**		**-35%**		**-9%**	

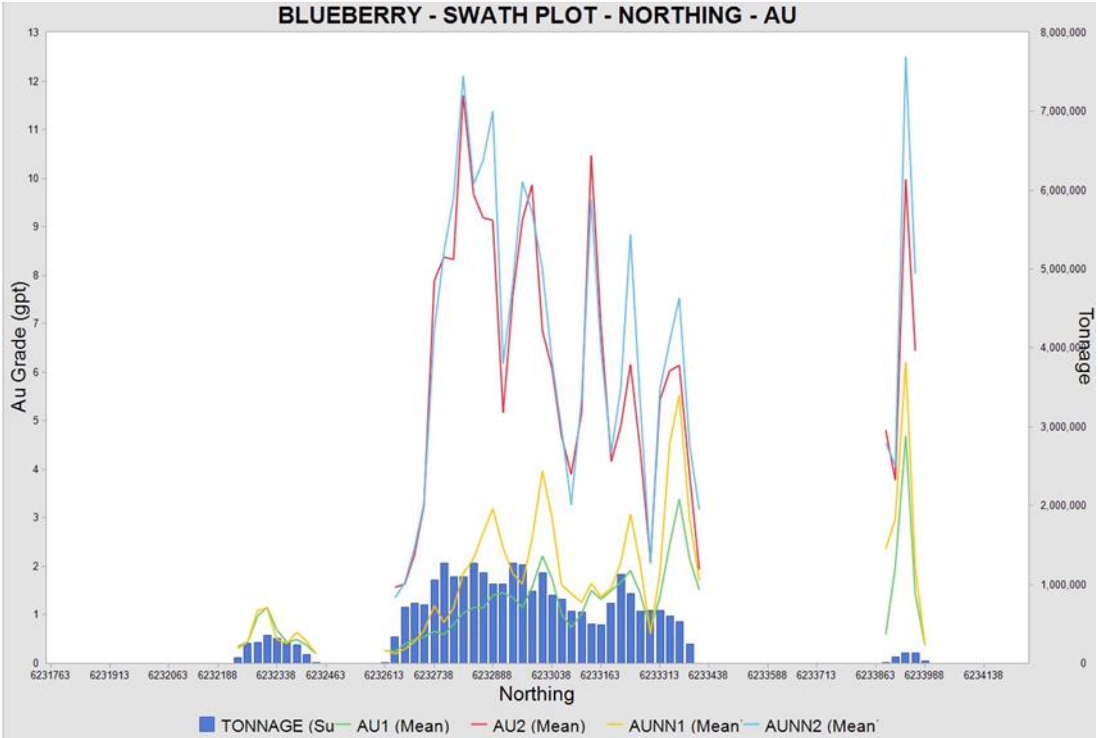


Figure 14-9: Swath Plot Au – Comparison of Interpolation Methods – Blueberry Domains for Open Pit (LG domains) and for Underground (HG domains) – Northing

(Source: MMTS, 2025)

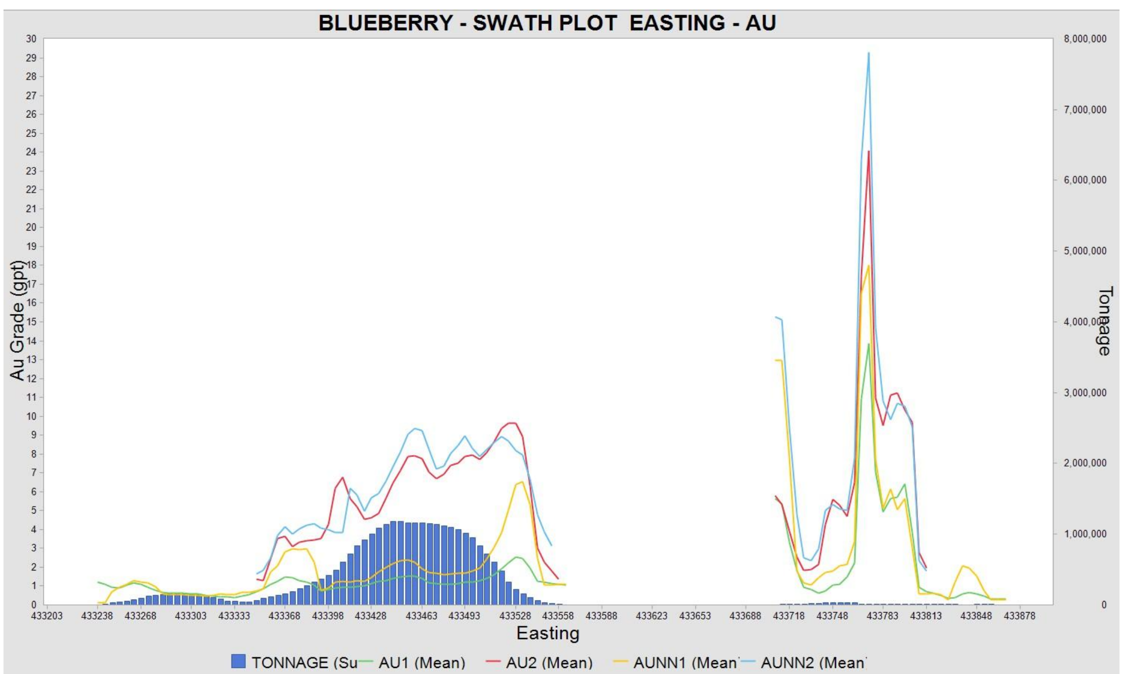


Figure 14-10: Swath Plot Au – Comparison of Interpolation Methods – Blueberry Domains for Open Pit (LG domains) and for Underground (HG domains) – Easting

(Source: MMTS, 2025)

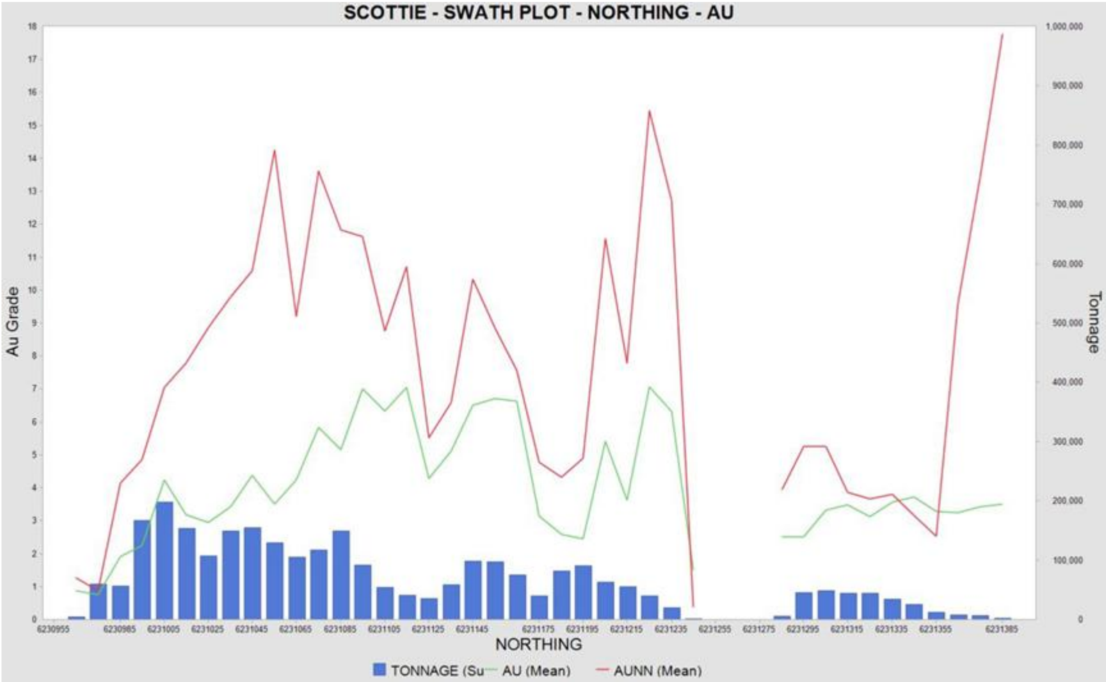


Figure 14-11: Swath Plot Au – Comparison of Interpolation Methods – Scottie Domains – Northing

(Source: MMTS, 2025)

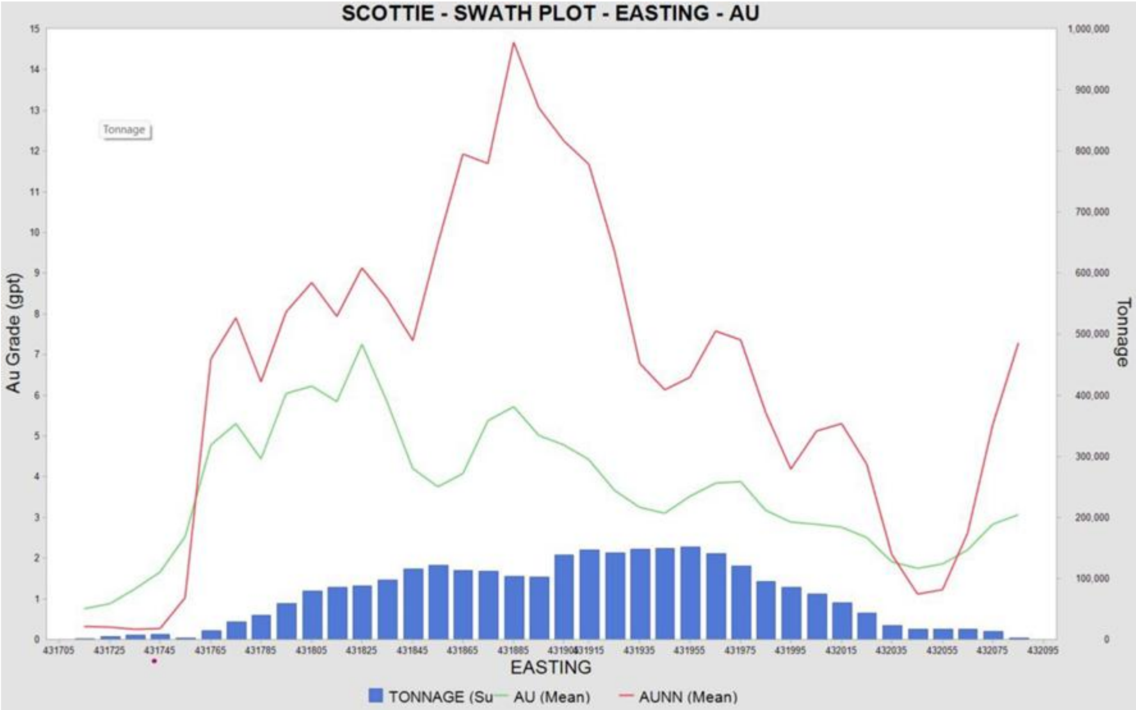


Figure 14-12: Swath Plot Au – Comparison of Interpolation Methods – Scottie Domains – Easting

(Source: MMTS, 2025)

14.9 Visual Validation

A series of E-W, N-S sections and plans have been used to inspect the modelled block grades with the original assay data. Figure 14-12 through Figure 14-14 illustrate the Au grade comparisons for each of the resource areas.

Plots throughout the model confirmed that the block model grades corresponded well with the assayed grades.

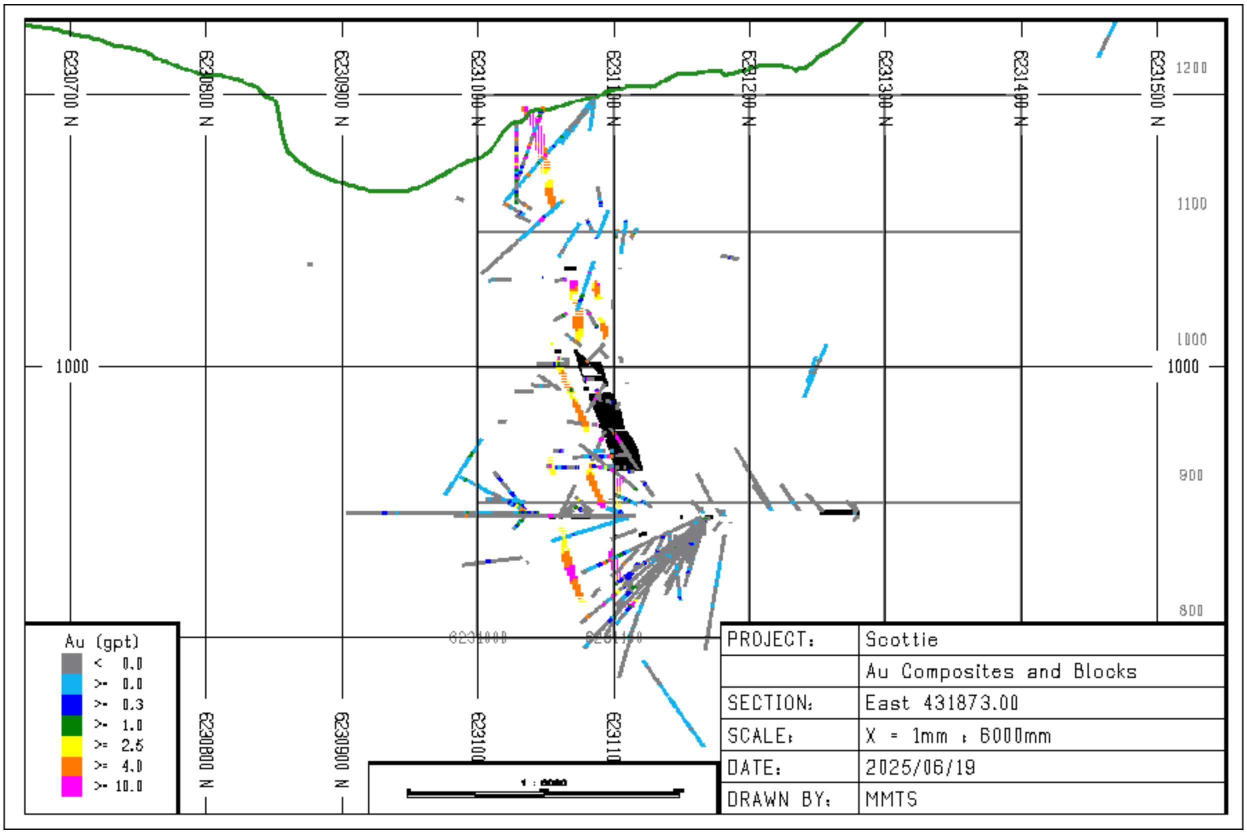


Figure 14-13: E-W Section – Comparing Au Grades for Block Model and Assay Data – Scottie

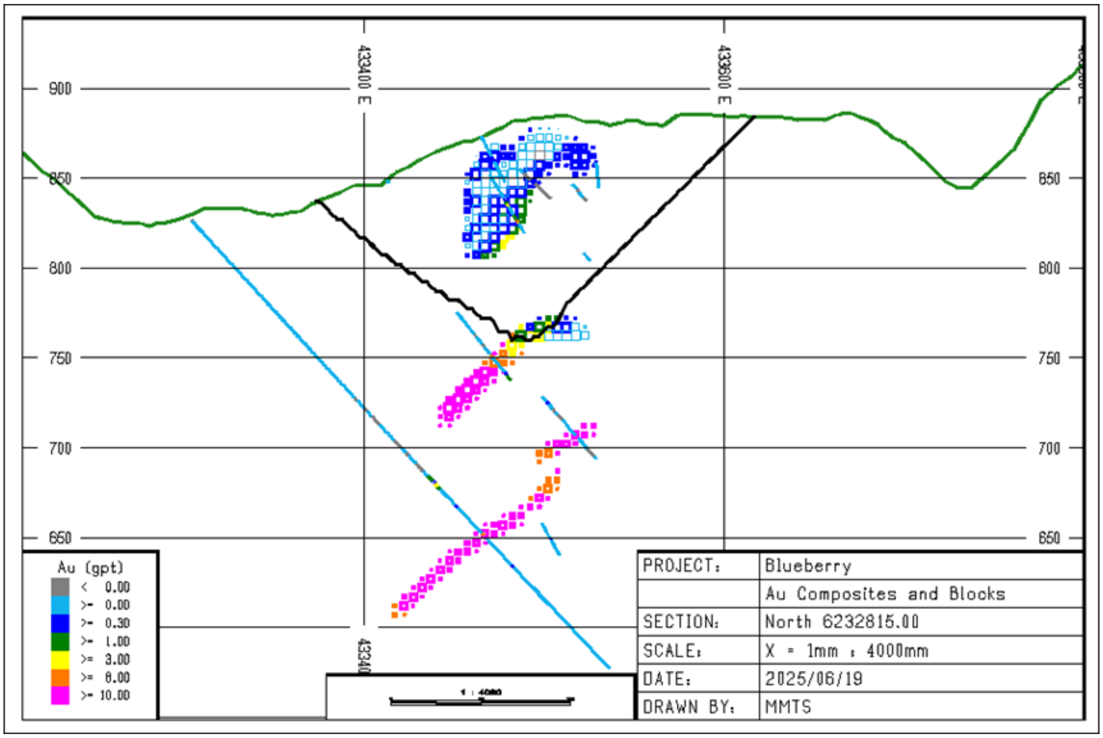


Figure 14-14: E-W Section Comparing Au Grades for Block Model and Assay Data – Blueberry

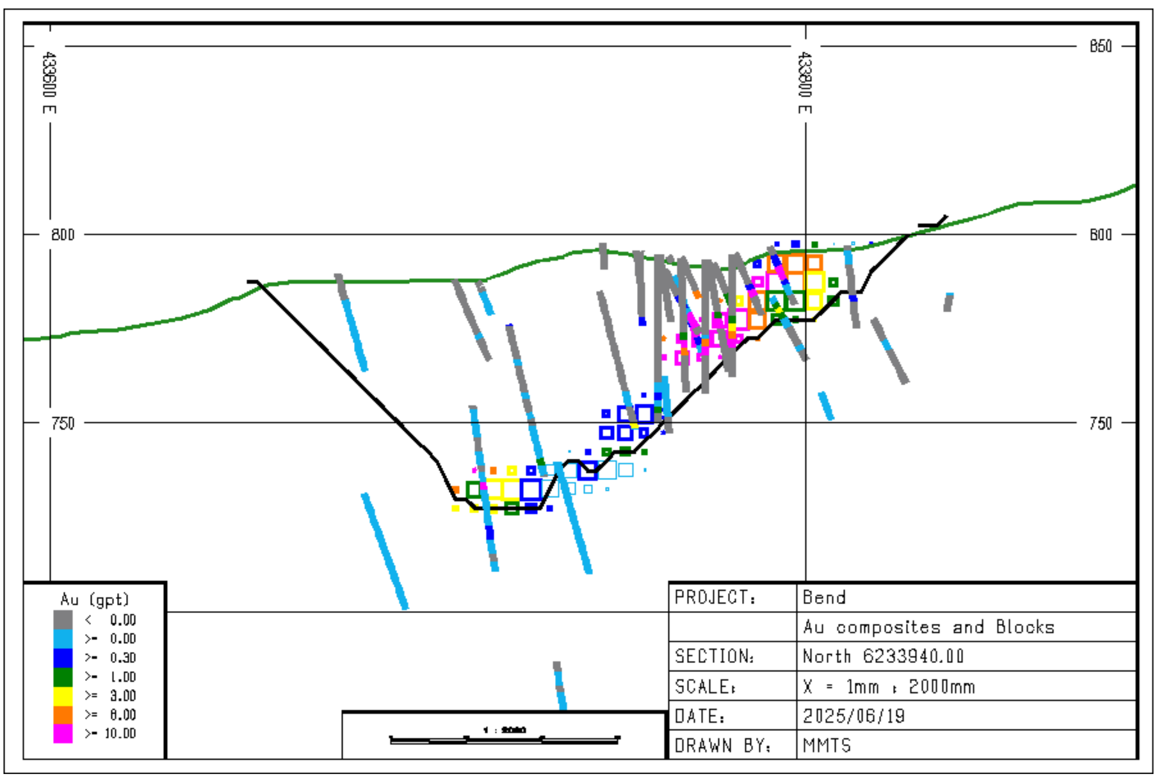


Figure 14-15: N-S Section – Comparing Au Grades for Block Model and Assay Data – Bend

14.10 Reasonable Prospects of Eventual Economic Extraction

The resource confining pit and/or underground shapes defines a boundary for continuous mineralization with suitable grades and with a reasonable expectation that an engineered plan will produce an economic plan. The net smelter return calculation for both the open pit and underground resources as well as the metallurgical recoveries are summarized in Table 14-11.

Lerchs-Grossman pits were run for each deposit using the following parameters:

- Pit slopes of 45 degrees
- Mining costs of CDN$3.00/t for both mineralized material and
- Processing and general and administrative (G&A) costs of CDN$24.00/tonne

Using a recovery of 90% au and smelter terms as outlined in the Notes to the Resource Table and summarized in Table 14-11, the open pit Au cutoff value is 0.31 g/t. The base case cutoff used for the resource is 0.70 g/t Au, which more than covers the Processing + G&A costs and mining costs.

Underground mining costs are based on comparables and are summarized in Table 14-12. The base case cutoff for the underground portion of the resource is 2.5 g/t Au which more than covers the Processing + G&A and mining costs.

Table 14-11: Economic Inputs and Metallurgical Recoveries

Parameter	Value	Units
Gold Price	$2,000.00	US$/Oz
Gold Payable	95.0%	%
Gold Refining	8.00	US$/oz
Gold Offsites	77.50	US$/WMT
Royalty	3.00%	%
Net Smelter Gold Price	54.646	US$/g
Gold Process Recovery	70%	%

Table 14-12: Underground Mining Costs

Item	Units	Cost (CDN)
UG Mining Cost	CDN$/t processed	$78.00
UG Development & Sustaining Capex	CDN$/t processed	$90.00
Processing and G&A	CDN$/t processed	$24.00
Total Cost - Underground	**CDN$/t processed**	**$198.00**

The pit delineated resource is given in Table 14-1 cut-offs with the base case cut-off of 2.5 g/t Au for underground and 0.7 g/t Au for the open pit. Process recoveries, as well as mining, processing and offsite costs have been applied in order to determine that the pit resource has a reasonable prospect of economic extraction.

14.11 Statement on Prospect of Economic Extraction

The QP is of the opinion that all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work.

14.12 Factors that may Affect the Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource estimate include:

- Commodity price assumptions
- Metal recovery assumptions
- Mining and processing cost assumptions

There are no other known factors or issues known to the QP that materially affect the estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors.

14.13 Risk Assessment

A description of potential risk factors is given in Table 14-13 along with either the justification for the approach taken or mitigating factors in place to reduce any risk.

#	Description	Justification/Mitigation
1	Classification Criteria	Classification is considered Inferred.
2	Gold Price Assumptions	Based on three-year trailing average (Kitco, 2025)
3	Capping	CPPs, Grade comparisons and swath plots show model validates well
4	Process and Mining Costs	Based on comparable projects in northern BC

15.0 MINERAL RESERVE ESTIMATE

There is no Mineral Reserve estimate for the Project as it is currently at a PEA stage. CIM defines a Mineral Reserve as:

"A Mineral Reserve is the economically mineable part of a measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study."

16.0 MINING METHODS

The Project is planned as a combined open pit and underground mining operation. Open pit development is expected to use a conventional truck-and-shovel method, whereas underground development will be based on longitudinal longhole stoping. A nominal average production rate of approximately 1,000 tonnes per day (tpd) has been assumed for the combined operation.

For the base case scenario, run-of-mine (ROM) mineralized material will be stockpiled and subsequently processed using an ore sorter. The ore-sorter product will be shipped directly to overseas markets. This mining and processing strategy has been applied to both the open pit and underground components and forms the basis of the preliminary economic assessment (PEA).

The Blueberry deposit was assessed as a combination of open pit and underground mining. The Scottie deposit, which was previously mined using shrinkage stoping ending 40 years ago, was assessed as underground only.

16.1 Open Pit Mine

Open pit mining will be carried out as a conventional truck and shovel operation. Mining is assumed to be done by contractors.

16.1.1 Block Model

The block model was based on the geological block model. The extents of the model are shown in Table 16-1 and the principal variables are shown in Table 16-2. The resource block model contained only mineralized blocks.

Table 16-1: Principal Variables of the Block Model

Dimension	Minimum	Maximum	Block Size (m)	No. of Blocks
X	432,400	434,100	5	340
Y	6,231,750	6,234,250	5	500
Z	200	1250	5	210

Table 16-2: Principal Variables of the Block Model

Variable	Description
AU1	Gold grade in g/t
DOM%1	Percent of the block within domain 1 (potentially mill feed)
SG1	Specific Gravity

Block Model Modifications

To prepare the block model for mine planning, waste was added to the model below topography with a SG of 2.84 and no gold grade.

16.1.2 Mining Limits

The topographic surface provided was limited to permitted boundary shown in Figure 4-2.

16.1.3 Geotechnical Information

The geotechnical review of the available preliminary data available for the Blueberry Zone indicates a generally competent rock mass, with three primary domains (Andesite, Contact Zone, Siltstone) defined for the PEA study. The three geotechnical domains provide a workable framework for the PEA. The preliminary slope design criteria presented in this PEA report are based on limited information and preliminary design assumptions. Further refinement is required as additional data become available during future phases of the Project:

- 20 m bench height
- 75 degree face angle
- 8.5 m catch bench width

16.1.4 Hydrogeological Information

No hydrogeological information was considered at this stage of the study. Data was collected in the summer of 2025 and will be used for further studies.

16.1.5 Dilution and Mine Recovery

Dilution and mining recovery have been estimated based on similar open pit mining operations and are shown in Table 16-3.

Table 16-3: Mining Recovery and Dilution

Parameter	Unit	Value
Mining Recovery	%	95
Dilution	%	5
Dilution grade	g/t	0

16.1.6 Pit Shell Optimization

A pit shell assessment was conducted using Datamine Studio NPVS, the resource model (blue25.csv), parameters, and constraints outlined in Table 16-4. Studio NPVS allowed for a number of scenarios to be assessed including a sensitivity on the gold selling price. The mining cost is representative of an assumed contractor operator scenario. Costs are in Canadian dollars, unless otherwise stated. All other units including tonnages are in metric, unless otherwise stated.

Table 16-4: Pit Shell Optimization Parameters

Parameter	Unit	Value
Mining Cost	$/t	6.95
Mining Dilution	%	5
Mining Recovery	%	95
G&A	$/t Plant Feed	15.6
Processing Cost[a]	$/t Plant Feed	7.1
Product Handling Cost[b]	$/t Plant Feed	129.2
Process Recovery	%	95.3
Selling Cost[c]	$/t.oz	460.8
Selling Price	$/t.oz	3545
Overall Slope	Degree	45

Note: [a]Processing includes crushing and costs associated to ore sorting. [b]Product handling cost includes all costs post ore sorting (i.e., stockpile, rehandle, transport to port, shipping, etc.). [c]Selling cost includes 2% royalties and average refining payabilities of 89%, which assumes a delivered grade of over 6 g/t. Ore sorting is assumed to reduce the mass of material required to be shipped by 46.3%, further discussed in Section 14. Payables are derived from the offtake agreement between Scottie and Ocean Partners.

The southern portion of the Blueberry resource model including an area adjacent to an avalanche gulley were excluded from the pit optimization. Preliminary analysis indicated that the lower-grade within these areas would adversely impact the pit payability.

The results of the pit shell analysis are summarized in Figure 16-1. Snowden Optiro selected the revenue factor (RF) 0.88 pit shell which represents a gold selling price of approximately 2240 $/oz as this shell, similar to the shells up to and including the RF 0.99 includes approximately 85 koz of gold post ore sorting. The RF 0.87 pit shell had about 4 koz less post ore sorting than the RF 0.88 pit shell. Details of RF 0.88 are presented in Table 16-5.

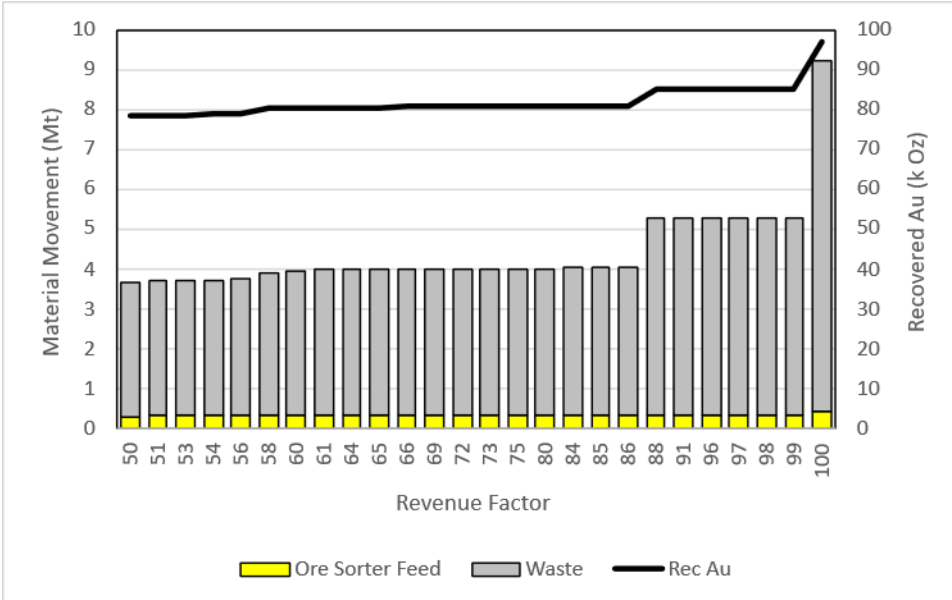


Figure 16-1: Pit Optimization Results by Revenue Factor

Table 16-5: Pit Optimization Selected Shell

RF	Waste (Mt)	Plant Feed (Mt)	Au (g/t)	Rec Au (k Oz)	Strip Ratio
88	4.90	0.36	7.67	85.2	13.5

16.1.7 Cut-Off

Cut-off grades were calculated using values listed in Table 16-4. An open-pit cut-off of 1.69 g/t Au was used shown in the below formulas.

$$Cut-off = \frac{(Product\ handling\ cost + Processing\ cost + G\&A\ cost) * Dilution}{(Selling\ Price - Selling\ cost) * Process\ Recovery/31.1035}$$

$$1.69\ g/t = \frac{(129.2 + 7.1 + 15.6) * 1.05}{(3545 - 460.8) * 0.953/31.1035}$$

16.1.8 Pit Design

A preliminary pit design, considering minimum mining width and ramp access was prepared to assess the pit shell, Section 16.1.3. A minimum mining width of 20 m was assumed and an overall ramp width, of 19 m was assumed with a maximum gradient of 10%, Figure 16-2.

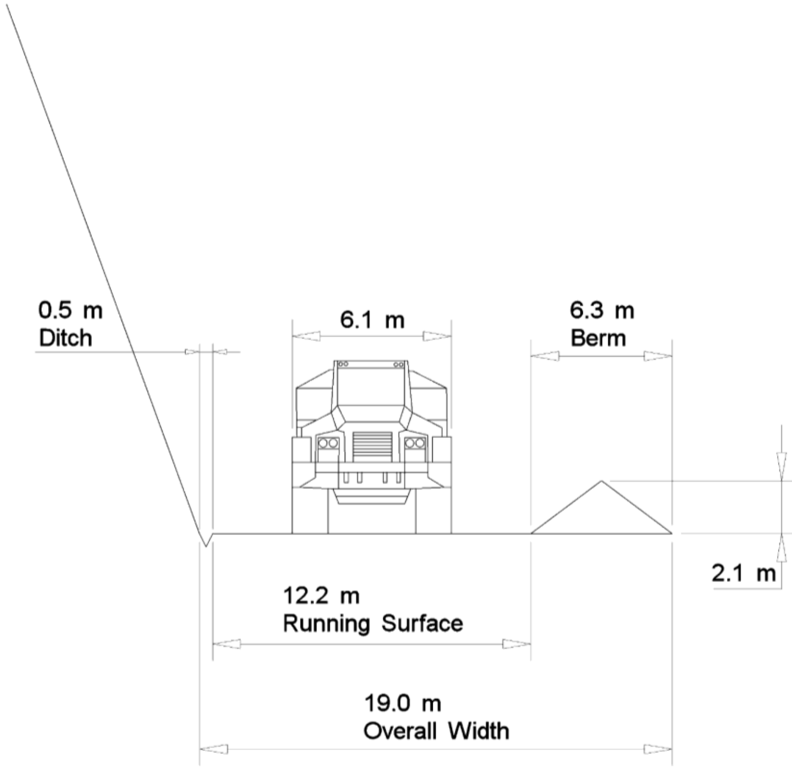


Figure 16-2: Cross-Section of Pit Ramp (Assuming no larger than a 100-t Haul Truck is Used)

16.1.9 Pit Phases

The pit was designed in four phases. Phase 1, named Bend since it is in an area where the current road has a switch-back (bend), and three phases in the Blueberry main area. The phases are shown in Figure 16-3. Mineral inventory by phase is shown in Table 16-6.

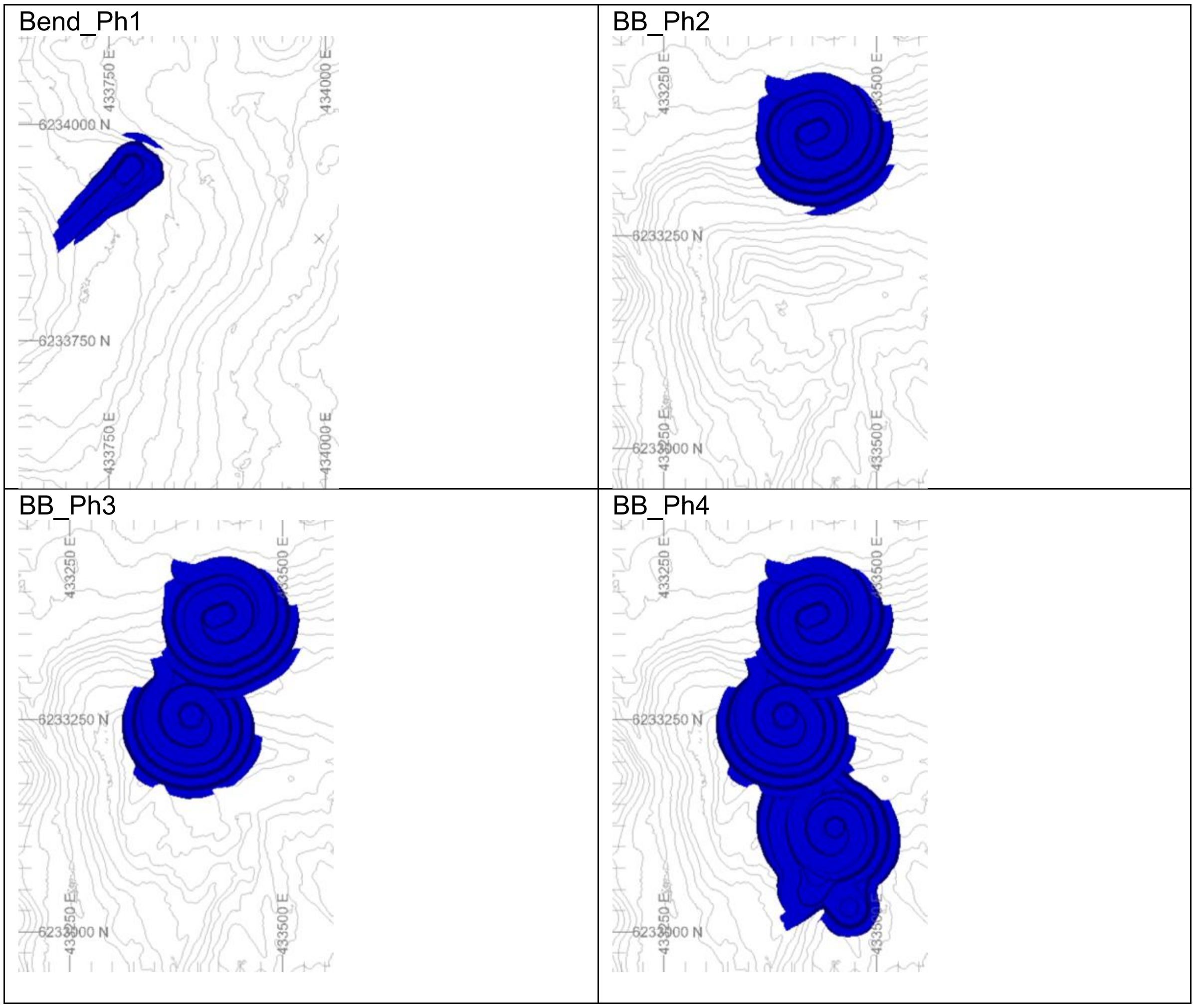


Figure 16-3: Mining Phases

Table 16-6: Mineral Inventory by Phase

Phase	Rock Waste (Mt)	Ore-Sorter Feed (Mt)	Au (g/t)	Rec Au (k Oz)
Bend_Ph1	0.25	0.03	9.56	8.5
BB_Ph2	1.83	0.16	8.10	38.8
BB_Ph3	1.85	0.08	5.12	12.1
BB_Ph4	1.93	0.06	9.23	16.3
Total	**5.86**	**0.32**	**7.71**	**75.7**

Recovered Au is post ore sorting.

16.1.10 Mining Equipment

Mobile mine equipment will be provided by the contractor. Due to the short mine-life, no replacement equipment will be required. Table 16-7 provides a breakdown of the potential major mobile mine equipment a contractor may use. The loading equipment selected allows for sufficient production capability along with selectivity.

Table 16-7: Open Pit Mining Equipment

Equipment	Size	Maximum Required
Haul Truck	100 t	4
Excavator	5 lcm bucket	1
Wheel Loader	5 lcm bucket	1
Production Drill	150 mm drill hole	1
Grader	250 Hp	1
Bulldozer	450 Hp	2
Water Truck	75,000 L	1

Note: Equipment contractor selects may be different from the units selected for this study.

16.1.11 Open Pit Sequence

The open pit mining sequence was scheduled using Datamine Studio NPVS and setting operational constraints such as maximum vertical development per period and goal of 1000 tpd ROM to the ore sorter. The areas mined per quarter are shown in Figure 16-4 (refer to Figure 16-3 for distance scale, location and orientation). The maximum vertical advance was limited to 30 m/quarter in a mining area.

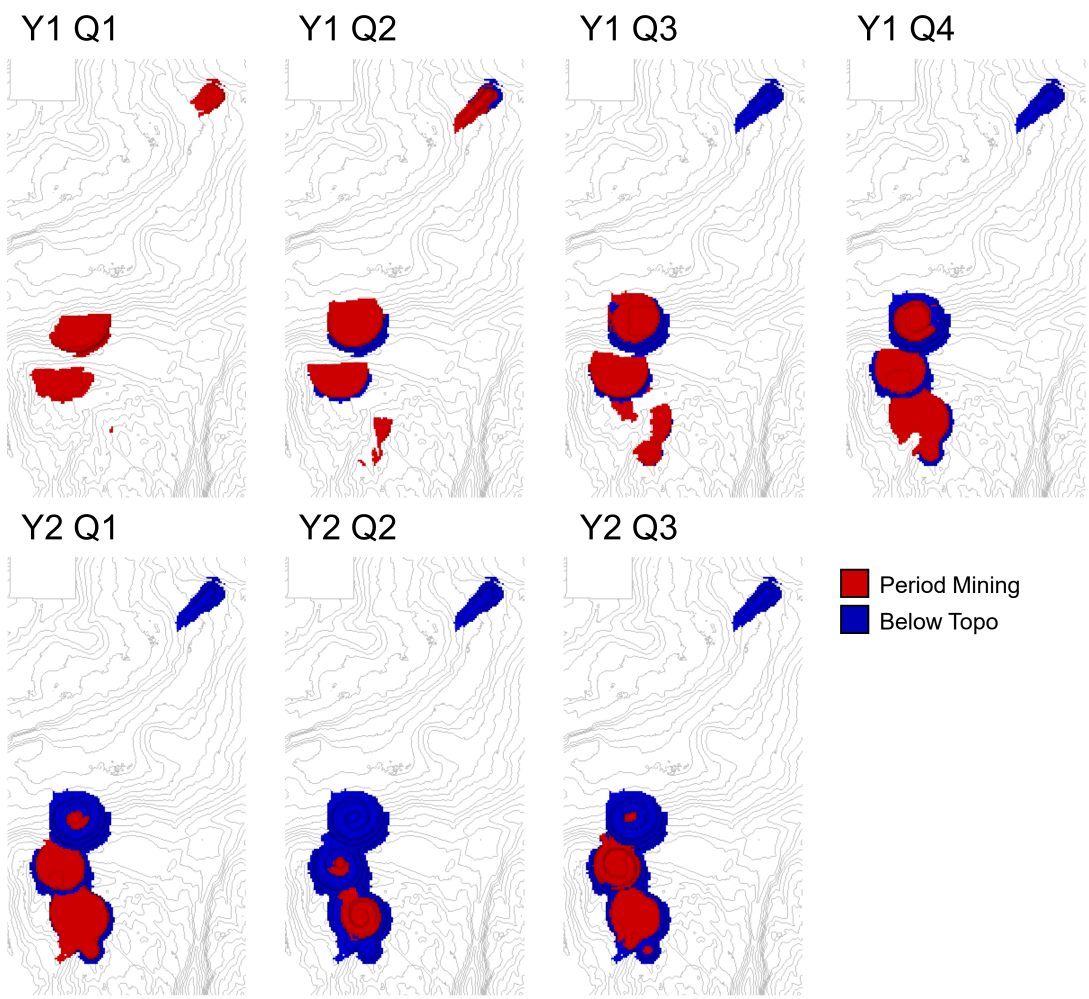


Figure 16-4: Blueberry Open Pit Mining Sequence

16.2 Underground Mining

The underground deposit consists of two separate areas which will be mined using longhole stoping method using retreat mining in a longitudinal stope orientation on 20 m level spacing.

16.2.1 Block Model

Two separate block models exist for Blueberry underground and Scottie underground. Both models were subcelled by Moose Mountain from the resource models with smaller blocks within the mineralized zone shapes to allow for more selective underground mining. The frameworks for the models are shown in Table 16-8 and Table 16-9 and the principal variables are shown in Table 16-10 and Table 16-11. The resource block model contained only mineralized blocks.

Table 16-8: Dimensions of the Blueberry Underground Block Model

Dimension	Minimum	Maximum	Parent Block Size (m)	Minimum Block	No. of Blocks
X	432,400	434,100	5	0.5	340
Y	6,231,750	6,234,250	5	0.5	500
Z	200	1250	5	0.5	210

Table 16-9: Dimensions of the Scottie Underground Block Model

Dimension	Minimum	Maximum	Parent Block Size (m)	Minimum Block	No. of Blocks
X	431,700	432,100	2	0.25	340
Y	6,230,950	6,231,400	2	0.25	500
Z	770	1200	2	0.25	210

Table 16-10: Principal Variables of the Blueberry Underground Block Model

Variable	Description
AU2	Gold grade in g/t
SUBDM	Ore domain within the model
SG2	Specific Gravity
DX	Block dimensions Easting in m
DY	Block dimensions Northing in m
DZ	Block dimensions Elevation in m

Table 16-11: Principal Variables of the Scottie Underground Block Model

Variable	Description
AU	Gold grade in g/t
SUBDM	Ore domain within the model
SG	Specific Gravity
DX	Block dimensions Easting in m
DY	Block dimensions Northing in m
DZ	Block dimensions Elevation in m

To confirm the subcelling was an accurate modification on the resource models, a comparison at different cut-offs was performed between the block models and are shown in Table 16-12 and Table 16-13. Subcelling values were within 0.5% of the resource model and was determined to be suitable for mine planning.

Table 16-12: Subcelled Check Blueberry Underground Block Model

	Resource Model			Subcelled Model		
Cut-off	Mass (Mt)	Au (g/t)	Au (kOz)	Mass (Mt)	Au (g/t)	Au (kOz)
2.5	1.60	9.02	464	1.59	9.02	462
3.5	1.33	10.24	438	1.33	10.24	436
5.0	1.00	12.25	393	0.99	12.26	391

Table 16-13: Subcelled Check Scottie Underground Block Model

	Resource Model			Subcelled Model		
Cut-off	Mass (Mt)	Au (g/t)	Au (kOz)	Mass (Mt)	Au (g/t)	Au (kOz)
2.5	0.70	7.15	160	0.70	7.18	161
4.0	0.47	9.03	137	0.47	9.06	138

Block Model Modifications

To prepare the block model for mine planning, waste was added to the model below topography with an assumed SG of 2.84 and no gold grade.

Block Model Depletion

The resource model provided for the study was depleted using an estimated underground mined-out volume derived from historical operations. Depletion shapes were generated from archived underground plan view sections, with stopes assigned a uniform width of 15 m. Subsequent discussions between Scottie Resources representatives and employees familiar with the historical underground operation indicated that actual stope widths were less than 5 m. This suggests that the 15 m depletion applied provides an inherent offset of approximately 5 m beyond the true stope limits on either side.

The British Columbia Health, Safety and Reclamation Code for Mines (Ministry of Energy, Mines and Low Carbon Innovation, 2024) states:

> *6.25.4 No work shall be carried out within 30 m of abandoned or old workings, or any accumulation of water or unconsolidated material, or any other substance that may flow, unless the proposed work procedure has been approved by the manager.*

For the Scottie deposit, it has been assumed that historical stopes can be backfilled, thereby allowing adjacent mining activities. Under this assumption, the current depletion methodology is considered acceptable for the level of mine planning undertaken in this study.

Further work, including detailed surveys and geotechnical analyses, will be required in subsequent phases of study to confirm the accuracy of historical depletion volumes and to ensure compliance with regulatory and safety requirements.

16.2.2 Mine Access

The mine will be accessed by a ramp decline due to the shallow nature of the mines, lower capital cost, existing drifts, and difficult access directly above Scottie deposit.

Decline Construction

The main decline will be driven using conventional drill and blast with an average advance rate of 4.1 m per day (m/d). This advance rate was assumed considering good conditions; however, the rate could be significantly reduced in wet or poor ground conditions where grouting and additional supports are required. A typical cross section is shown in Figure 16-5. Grouting will be used to control water inflows if required. The decline will be 5.5 m x 5.5 m.

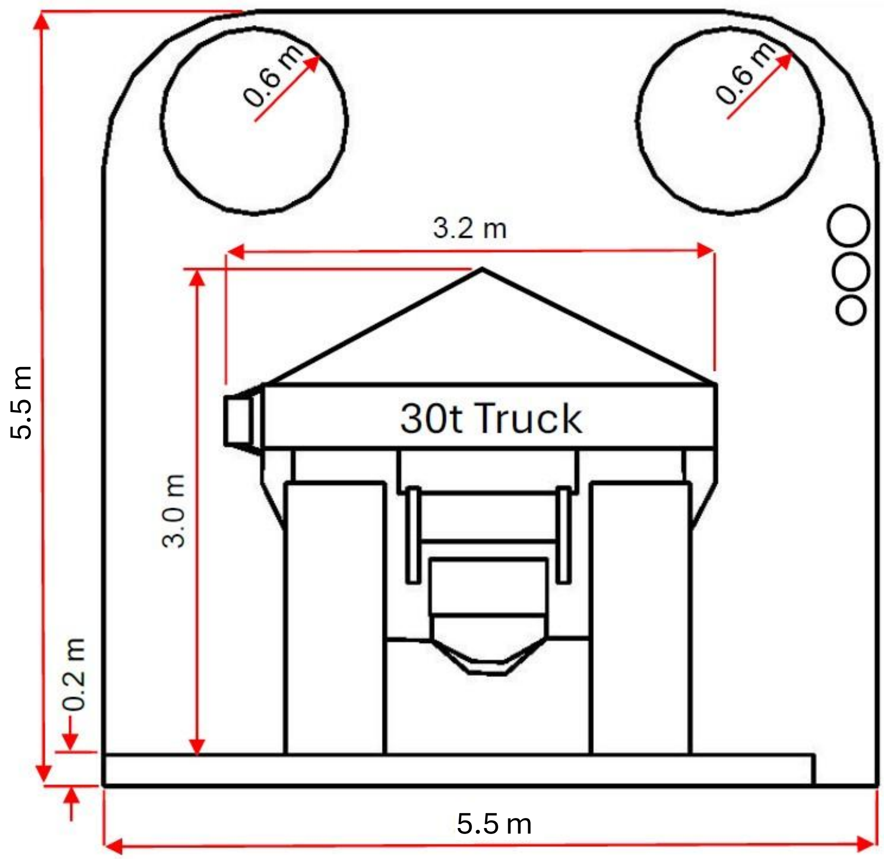


Figure 16-5: Cross Section of Main Decline

Lateral Development

The lateral development will be driven 5 m wide by 5 m high. The drift size was selected to provide sufficient clearance for a 30-tonne truck. The drift height is required to provide adequate clearance for installing ventilation ducts and providing sufficient ventilation airflow volumes for truck haulage operation during the production period. Safety bays will be required with this configuration.

Ventilation Raise

The ventilation exhaust raise will be excavated by contractors using a raise bore. The ventilation shaft diameter will be 3.5 m. A detailed ventilation simulation was not conducted; therefore, the dimensions of the vent raises will need to be confirmed in future studies.

16.2.3 Backfill

A detailed backfill plan was not prepared for this initial assessment; however, the plan is to use loose rock backfill from waste rock. A detailed backfill plan will be determined in future studies.

16.2.4 Stoping

s ing Datamine's MS software with the block model described in 17.2.1, longhole mining was assessed using stope parameters presented in Table 16-14.

Table 16-14: Longhole Stope Parameters

Parameter	Unit	BB	SCT
Stope Strike Length	m	10	10
Minimum Stope Width	m	3	3
Maximum Stope Width	m	20	20
Minimum Pillar Length	m	10	10
Stope Height	m	20	20
Minimum Dip Angle	degree	50	50

Following stope and mine development design the result was run through STOPEMAX which optimizes the design by adjusting cut-off grades within levels and removing stopes that are uneconomic due to development costs.

16.2.5 Ground Support

Due to the limited geotechnical information, a basic assumption for all tunnels has been used, which is poor to fair ground conditions using the Q rock mass system by the Norwegian Geotechnical Institute (NGI). Development is expected to have spans of around 5 m with an excavation support ratio (ESR) between 1.3 (access tunnels) and 2 (vertical shafts). The required rock support classification is shown in Figure 16-6 with the support suggestions being split between 1 (spot bolting) and systematic bolting and shotcrete (5 cm to 6 cm).

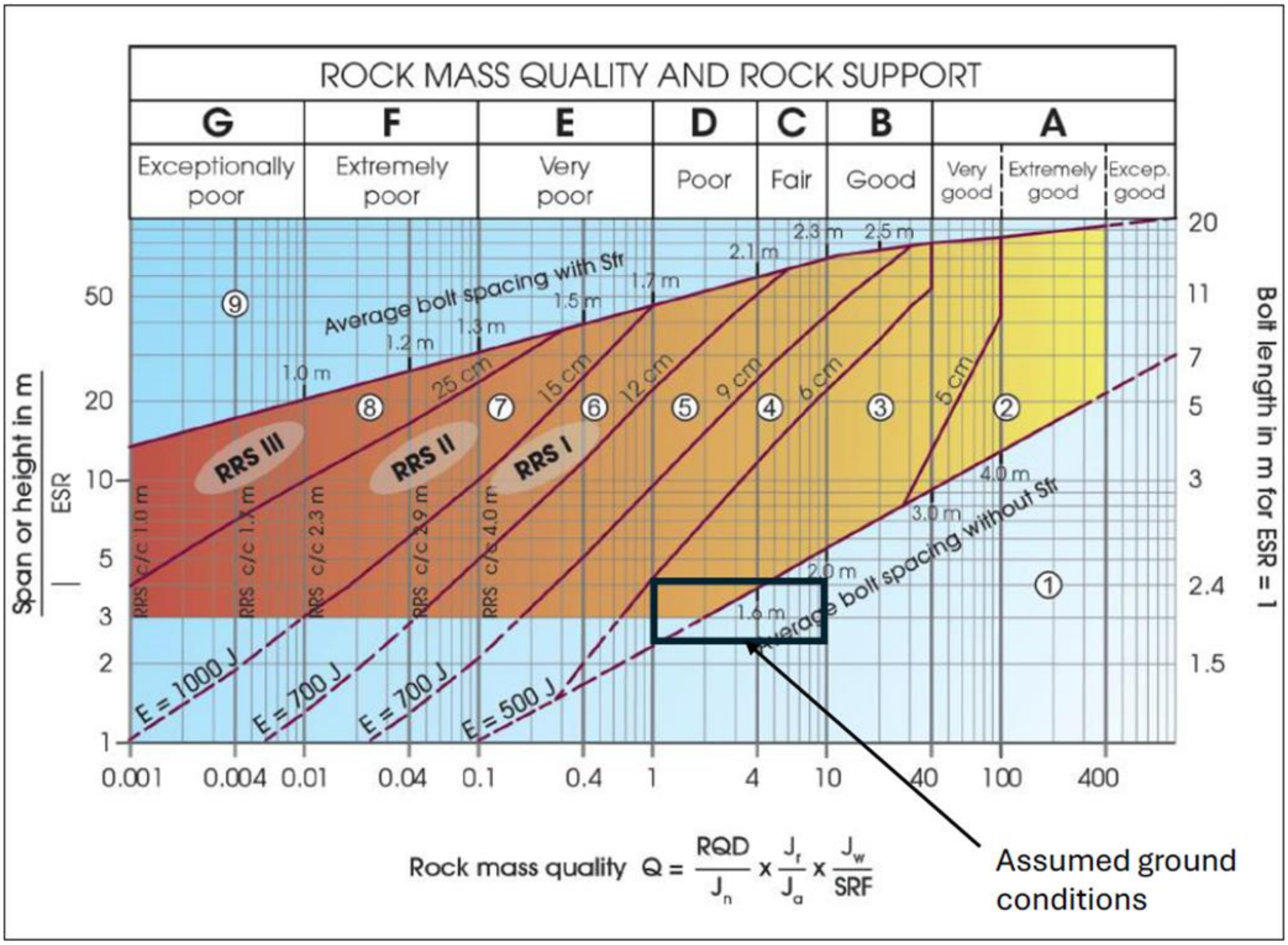


Figure 16-6: NGI Empirical Rock Mass Support Chart (Norwegian Geotechnical Institute, 2022)

Based on the Snowden QP’s previous underground experience, systematic bolting (1.8 m long on a 1.2 m by 1.2 m pattern) and shotcrete (5 cm to 6 cm) for all areas of the mine was assumed.

16.2.6 Cut-Off

Separate cut-offs were used for development tonnage and stoping tonnage. Development tonnage uses the same cut-off as open pit mining as described in Section 16.1.7. A stoping cut-off includes the stope mining cost and additional dilution using the parameters shown in Table 16-15. Costs are in Canadian dollars, unless otherwise stated.

Table 16-15: Underground Parameters

Parameter	Unit	Value
Stoping Cost	$/t	92.7
Mining Dilution	%	5
Mining Recovery	%	95
Average Mining Width	m	14.0
Sidewall Dilution	m	0.3
G&A	$/t plant feed	15.6
Processing Cost[a]	$/t plant feed	7.1
Product Handling Cost[b]	$/t plant feed	129.2
Process Recovery	%	95.3
Selling Cost[c]	$/t.oz	460.8
Selling Price	$/t.oz	3545

Note: [a]Processing includes crushing and costs associated to ore sorting. [b]Product handling cost includes all costs post ore sorting (i.e. stockpile, rehandle, transport to port, shipping, refining, etc.). [c]Selling cost includes 2% royalties and refining payabilities of 89%. Ore sorting is assumed to reduce the mass of material required to be shipped by 46.3%, further discussed in Section 13. Payables are derived from the agreement between Scottie and Ocean Partners.

A stoping cut-off was calculated at 2.8 g/t; however, a final cut-off of 2.9 g/t was used to increase mined grade.

$$Cut-off = \frac{(Product\ handling\ cost + Processing\ cost + G\&A\ cost + stoping\ cost) * Dilution}{(Selling\ Price - Selling\ cost) * Process\ Recovery/31.1035}$$

$$2.8\ g/t = \frac{(129.2 + 7.1 + 15.6 + 92.7) * (1.05 + \frac{0.3 * 2}{14})}{(3545 - 460.8) * 0.953/31.1035}$$

16.2.7 Mining Equipment

The mining contractor will provide the required mining equipment depending on the mining schedule and will generally consist of a single development and a single production crew. Due to the short mine life, major equipment is not required. Table 16-16 provides a breakdown of the major mobile mine equipment at its peak requirement.

Table 16-16: Underground Mining Equipment

Parameter	Maximum Required
LHD	3
Haul Truck	4
Drills	3
Bolter	1
ANFO Charger	1

Additional support equipment such as scissor lifts, personnel carriers, supervisor vehicles, forklifts, etc. are also included in the mine plan.

Fixed Mining Equipment

<u>Fans</u>

To properly ventilate the mine, two main types of fans will be required, including:

- The intake fans, 200 kW fan located on surface at the main portal.
- Auxiliary fans each rated for 80 kW to provide proper circulation around the mine workings.

Scottie and Blueberry will each have their own intake fans; auxiliary fans will be shared as required.

<u>Heating</u>

Heating will be required to raise intake air to a minimum of 3°C during the winter months using a natural gas direct fired heating system with a minimum power rating of 7 MW. An assumed consumption of approximately 1.2 million m^3 of natural gas per year.

<u>Refuge System</u>

Each of the mining areas (Blueberry and Scottie) will have its own portable refuge station. A total of three refuge stations were included in the initial cost estimate.

16.2.8 Ventilation

The ventilation system will be established using the main decline for fresh air supply and a dedicated exhaust shaft. A detailed ventilation study has not been completed and is recommended for advancing the Project.

Airflow Assumptions

The airflow requirements have been determined based on standard operating procedures in Canada which use approximately 3.6 m^3/min. Total maximum airflow requirements are shown in Table 16-17. An assumption of 10% leakage was assumed.

Table 16-17: Airflow Assumptions

Category	Maximum Airflow Requirements (m^3/s)
Equipment	160
Active Levels	20
Facilities	10
Leakage	19
Total	**209**

Airflow Velocities

Airflow requirements will result in air velocities up to approximately 5 m/s in the main declines which is within standard design criteria. Detailed ventilation modelling will be required to confirm airflow requirements and velocities.

16.2.9 Secondary Egress

Secondary personnel egress will be installed in the exhaust raise at Scottie and a separate escape way raise in Blueberry that uses a self-contained modular ladderway attached to the raise liner.

16.2.10 Production Rates

Development rates were applied based on horizontal and vertical development. Vertical rates were applied to all vertical and raise bore development. Table 16-18 summarizes the development rates applied in the schedule.

Table 16-18: Development Rates

Parameter	Dimensions (m)	Rate (m/mth)	Cost ($)
Decline	5.5 x 5.5	123	6,500
Level Access	5.0 x 5.0	90	5,000
Ore Drive	5.0 x 5.0	65	5,000
Raisebore	3.5	90	9,000
Raise	4.0 x 4.0	90	4,000

16.2.11 Mine Optimization

Once a development plan was completed for each deposit, costs were assigned and StopeMax was utilized to determine if stopes are still economically viable given the development costs. Figure 16-7 shows the stopes and development that were removed through optimization in pink.

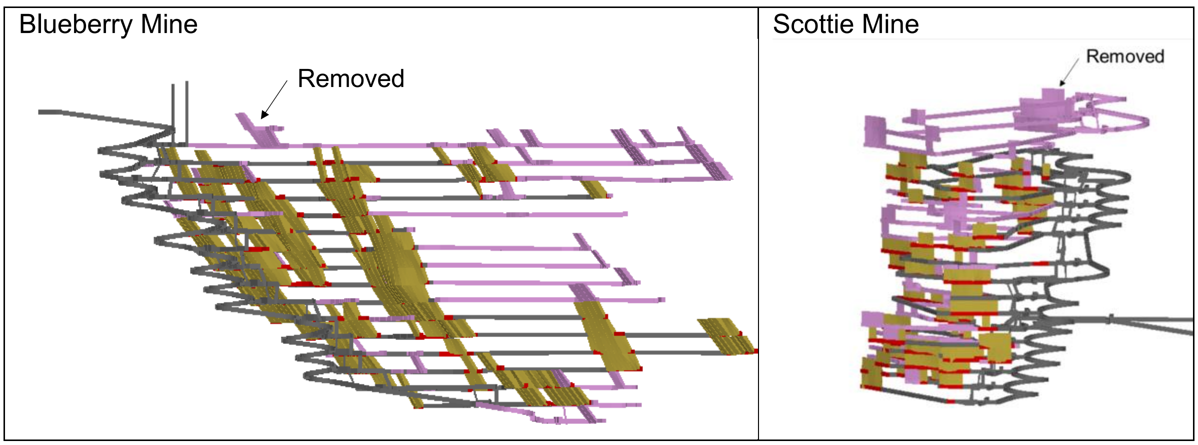


Figure 16-7: Removed Stopes

16.2.12 Mining Sequence

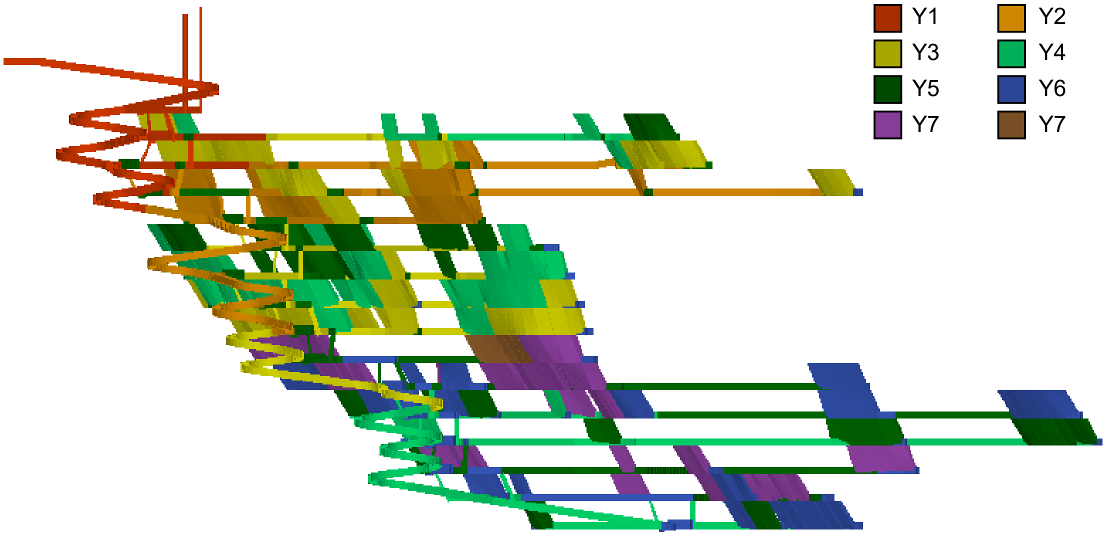


Figure 16-8: Mine Sequence Blueberry UG

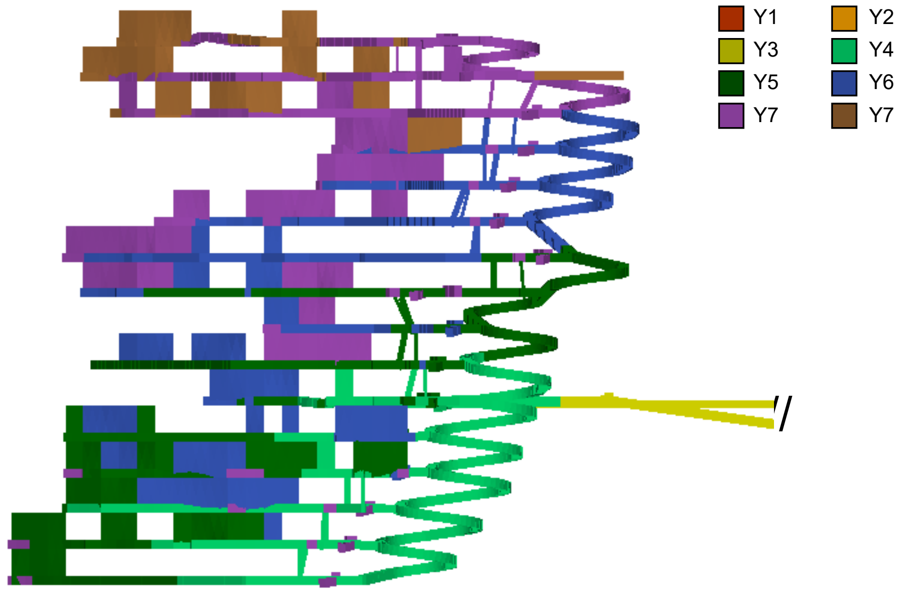


Figure 16-9: Mine Sequence Scottie UG

16.3 Mining Labour

Labour requirements for the integrated open pit and underground operations have been developed on a first principal basis. The majority of personnel will work on two-weeks-on two-weeks-off rotation basis, alternating between day and night shift. Maximum labour requirements are summarized in Table 16-19 and shown on a quarterly basis in Figure 16-10.

Table 16-19: Mining Labour Requirements

Department	Maximum Labour Requirements
Operations Staff	8
Underground Operations	98
Open Pit Operations	40
Maintenance Staff	4
Maintenance Hourly	42
Technical Staff	22
Total	**214**

Note: [a]Total includes personnel for pit and two underground operations; however, the schedule dictates that the open pit will be completed when two underground operations are active. Scheduled maximum of 184 mining personnel (on payroll) would be required for the first years.

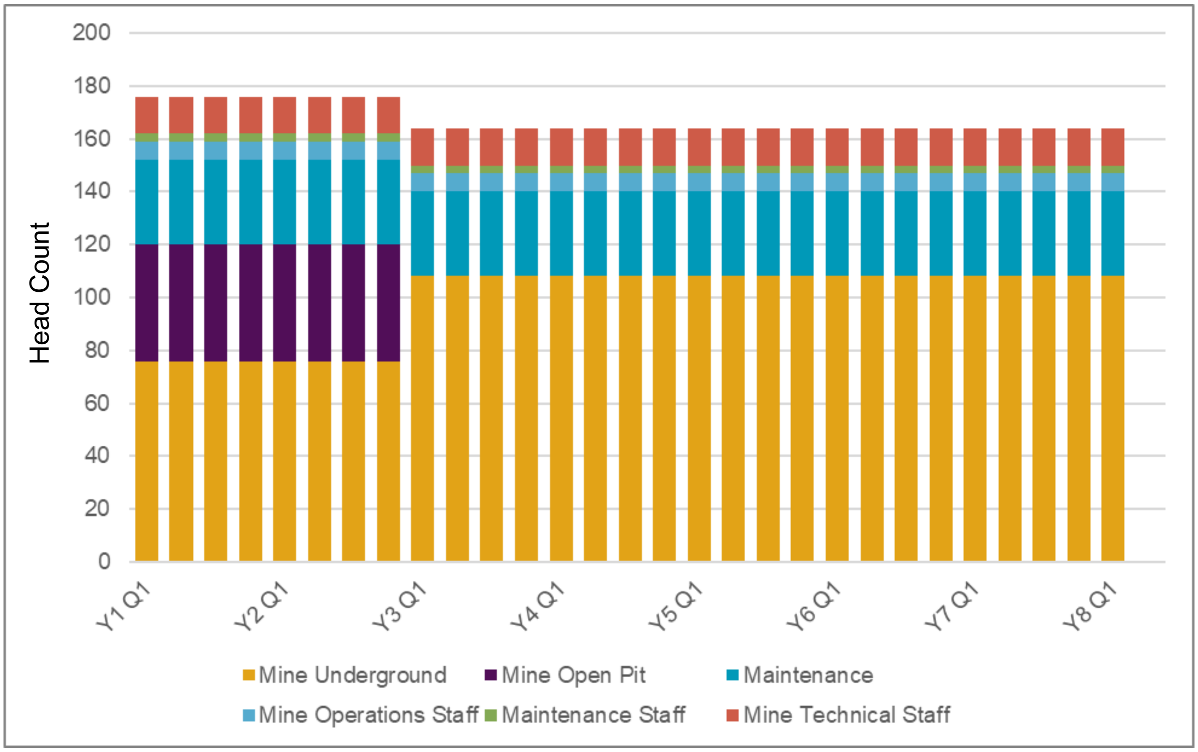


Figure 16-10: Personnel by Quarter

16.4 Mine Schedule

The annual mine schedule for both open pit (OP) and underground (UG) is shown in Table 16-20. Quarterly ore-sorter feed and underground development tonnes are shown in Figure 16-11 and Figure 16-12.

Table 16-20: Mine Schedule

Description	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Ore-Sorter Feed										
Total ore-sorter feed tonnage	(kt)	2,191	214	400	309	354	365	307	239	4
Grade – Au	(g/t)	6.86	7.7	8.2	7.7	6.6	5.9	5.3	6.7	5.3
Metal content – Au	(koz)	460	51	100	72	71	66	50	49	1
OP – Total										
Total tonnage	(kt)	5,862	3,600	2,261	0	0	0	0	0	0
Total waste tonnage	(kt)	5,541	3,417	2,124	0	0	0	0	0	0

Description	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Total ore-sorter feed tonnage	(kt)	320	183	137	0	0	0	0	0	0
Grade – Au	(g/t)	7.71	7.6	7.8	0.0	0.0	0.0	0.0	0.0	0.0
Metal content – Au	(koz)	79	45	34	0	0	0	0	0	0
UG – Total										
Total tonnage	(kt)	3,596	222	422	675	711	674	501	387	4
Total waste tonnage	(kt)	1,725	191	159	367	357	309	194	148	0
Total ore-sorter feed tonnage	(kt)	1,871	30	263	309	354	365	307	239	4
Grade – Au	(g/t)	6.71	8.32	8.38	7.65	6.56	5.90	5.30	6.74	5.34
Metal content – Au	(koz)	406	8	71	76	75	70	53	52	1
Blueberry ore tonnage	(kt)	1,421	30	263	309	294	233	190	102	0
Grade – Au	(g/t)	6.91	8.32	8.38	7.65	6.77	5.29	5.10	7.94	0.00
Blueberry metal content – Au	(koz)	316	8	71	76	64	40	31	26	0
Scottie ore tonnage	(kt)	450	0	0	0	60	132	117	137	4
Grade – Au	(g/t)	6.07	0.00	0.00	0.00	5.51	6.98	5.63	5.84	5.34
Scottie metal content – Au	(koz)	88	0	0	0	11	30	21	26	1
UG Development – Total										
Lateral development	(m)	25,985	2,507	2,884	4,879	5,588	5,171	2,782	2,173	0
Vertical development	(m)	5,732	290	733	810	1,224	1,032	949	679	0
Blueberry lateral development	(m)	13,422	2,507	2,884	2,850	2,796	2,385	0	0	0
Blueberry vertical development	(m)	3,501	290	733	767	887	329	301	195	0
Scottie lateral development	(m)	12,564	0	0	2,029	2,792	2,786	2,782	2,173	0
Scottie vertical development	(m)	2,231	0	0	44	337	704	647	484	0

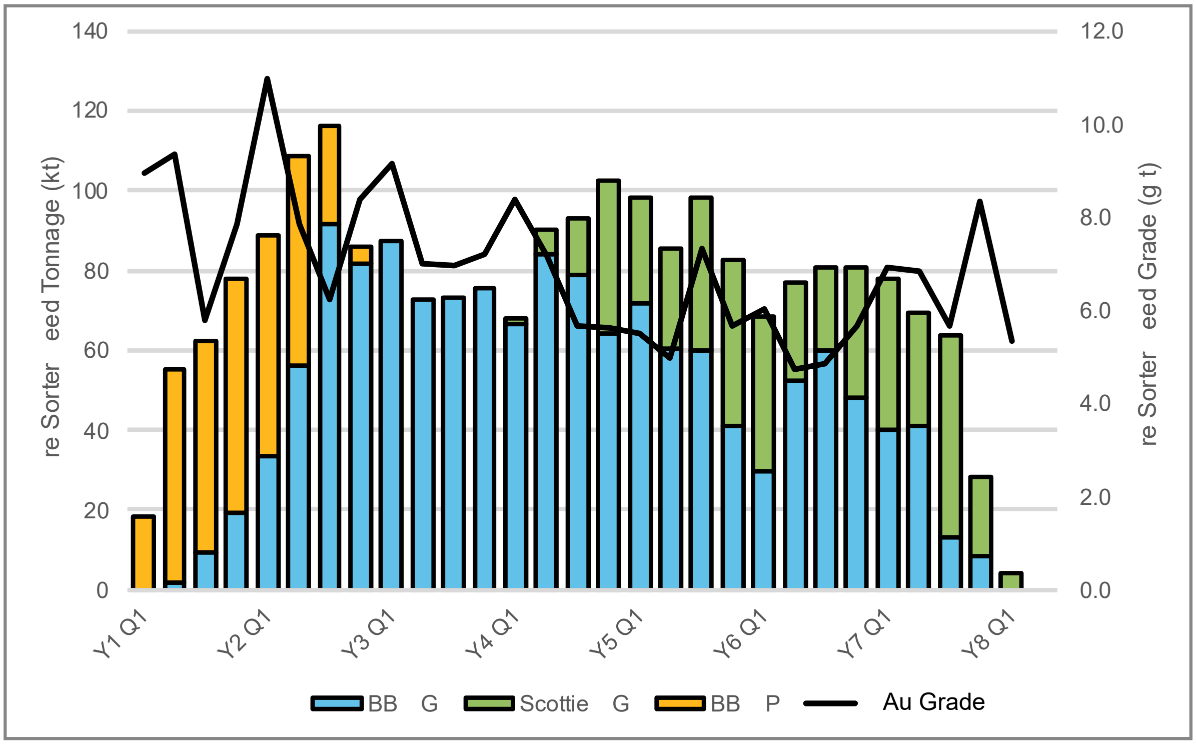


Figure 16-11: Quarterly Ore-Sorter Feed

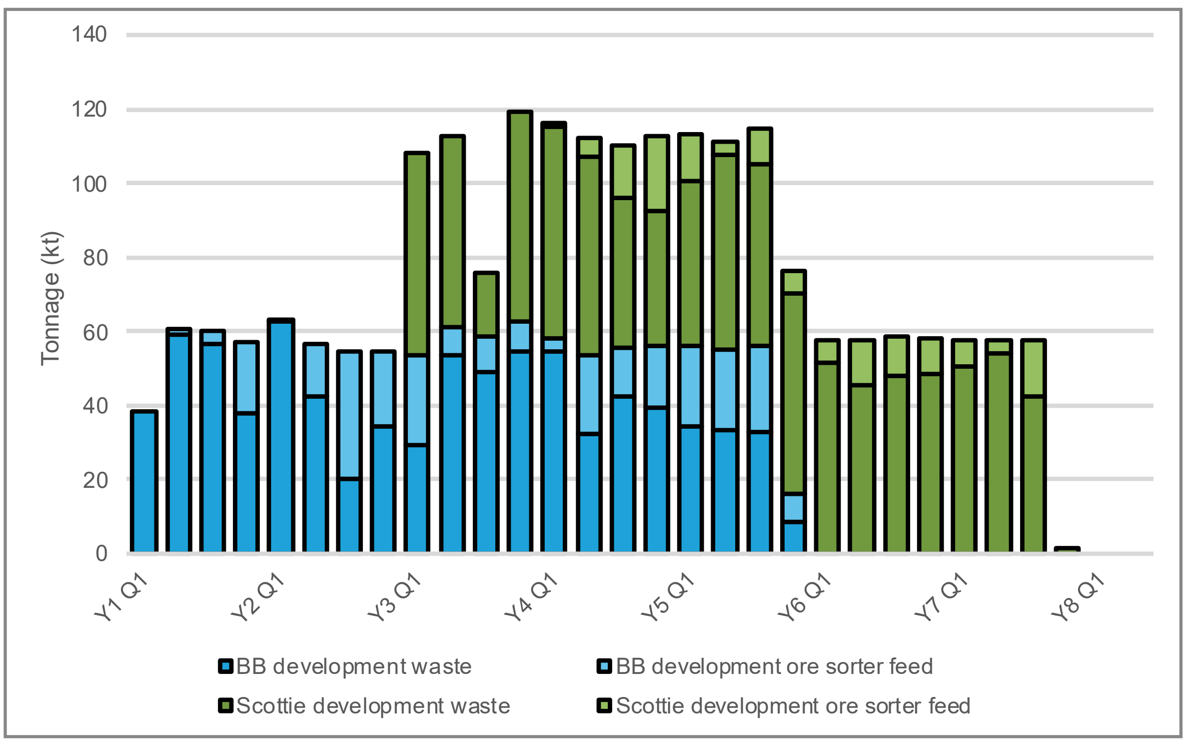


Figure 16-12: Quarterly Underground Development

16.5 Potential Ore-Sorter Feed

There is no Mineral Reserve estimate for the project. An initial assessment was completed, which included material classified as Inferred, which will be referred to as potential ore-sorter feed. There was no measured or indicated material in the project and no waste material has contained metal. The potential ore-sorter feed which includes all material above cut-off in the pit (Section 16.1.7), material from within the stopes plus select segments of development where total Au grade was above 1.7 g/t is shown in Table 16-21 through Table 16-23.

Table 16-21: Open Pit Potential Ore-Sorter Feed

Phase	Rock (Mt)	Ore Sorter Feed (Mt)	Waste (Mt)	Au (g/t)	Au Met to Mill (kOz)	Au Met Recovered (kOz)	Strip Ratio
Bend Ph1	0.25	0.03	0.23	9.56	9	8	7.8
BB Ph2	1.83	0.16	1.68	8.10	41	39	10.7
BB Ph3	0.18	0.08	0.11	5.12	13	12	1.4
BB Ph4	0.19	0.06	0.14	9.23	17	16	2.3
Total OP	**5.86**	**0.32**	**5.54**	**7.71**	**79**	**76**	**17.3**

Table 16-22: Underground Potential Ore-Sorter Feed

Phase	Rock (Mt)	Ore Sorter Feed (Mt)	Waste (Mt)	Au (g/t)	Au Met to Mill (kOz)	Au Met Recovered (kOz)
Scottie Stopes	0.31	0.31	0.00	6.54	66	63
Scottie Development	0.00	0.14	0.91	4.99	22	21
BB Stopes	1.15	1.14	0.01	7.06	260	248
BB Development	1.09	0.27	0.82	6.34	55	53
Total UG	**2.55**	**1.87**	**1.73**	**6.72**	**403**	**384**

Table 16-23: Combined Potential Ore-Sorter Feed

Phase	Rock (Mt)	Ore Sorter Feed (Mt)	Waste (Mt)	Au (g/t)	Au Met to Mill (kOz)	Au Met Recovered (kOz)
Total OP	5.86	0.32	5.54	7.71	79	76
Total UG	2.55	1.87	1.73	6.72	403	384
Total	8.41	2.19	7.27	6.87	482	460

17.0 RECOVERY METHODS

17.1 Introduction

The metallurgical test work detailed in Section 13, together with the mine plan described in Section 16, form the basis for the selection of recovery methods for the Project.

The feeds to the proposed preconcentration plant for the project are sourced from three distinct deposits: BBOP, BBUG, and SGMUG deposits. Mining will be conducted using a conventional truck-shovel open-pit method for BBOP, and longitudinal long-hole stoping underground mining method for both BBUG and SGMUG. Run-of-mine (ROM) plant feed is processed through single-stage crushing, operating in a closed circuit with a screen. The screen products—coarse and fine fractions—are treated using respective coarse and fine ore-sorters to produce separate concentrates. The concentrates are blended with fine fraction generated from the sorter feed screening and shipped offsite as the final product.

17.2 Flowsheet Development

The processing plant is designed to process the plant feed at a nominal throughput of 1,000 t/d, with a LOM average of 900 t/d, producing gold concentrate under a day-shift operation strategy. The LOM average plant feed grade will be 6.86 g/t Au, and the anticipated average recovery will be 94.7%, including the fines blended with the sorter concentrates. The LOM average annual concentrate production will be approximately 177,000 t/y at 11.5 g/t Au.

The processing plant will consist of the following:

- A primary crusher operating in closed-circuit with a triple-deck dry screen
- A fine product stockpile
- A double-deck wet screen to prepare the feeds for ore-sorters
- X-ray fluorescence sorters (coarse and fine) to produce final concentrate
- Belt filter to recycle wash water from the wet screen
- All other associated utilities required for plant operation

The simplified process flowsheet is shown in Figure 17-1 and detailed in the following sections.

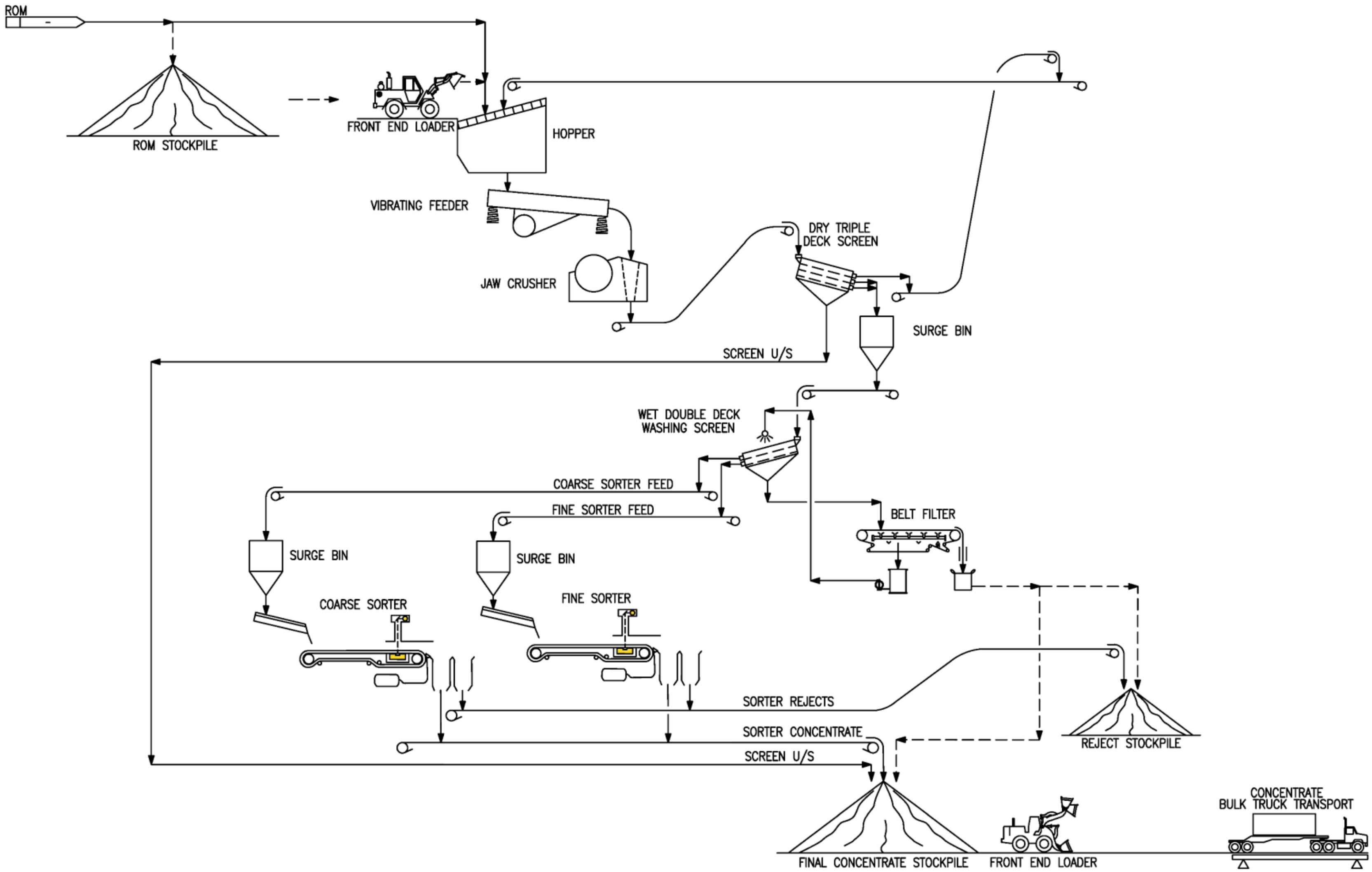


Figure 17-1: Simplified Process Flowsheet

17.3 Process Design Criteria

The major criteria used for the design of the processing plant are listed in Table 17-1.

Table 17-1: Major Plant Design Criteria

Description	Unit	Value
Operating Schedule		
Shift/Day	--	1
Plant Hours/Shift	h	12
Days/Year	days	365
Plant Feed		
Plant Throughput	t/d	1,000
Total Plant Feed (LOM)	Mt	2.19
Average Gold Grade (LOM)	g/t	6.86
Crushing		
Crushing Circuit Availability	%	70
Crushing Circuit Feed Rate	t/h	119
Fines Cut Size	mm	12.5
Fines Percent	%	21
Sorter		
Coarse Sorter Cut Size	mm	100 + 37.5
Fine Sorter Cut Size	mm	37.5 + 12.5
Sorter Concentrate Yield including Fines	%	56.5
Concentrate Average Gold Grade (LOM)	g/t	11.5

17.4 Process Description

The sorter feed preparation circuit consists of a jaw crusher operating in a closed circuit with a dry triple-deck primary screen, featuring apertures of 100 mm (4”), 37.5 mm (1.5”), and 12.5 mm (0.5”). Oversize material from the screen is recirculated to the jaw crusher for further size reduction. The coarse fractions from the primary screen (-100 +37.5 mm and -37.5 +12.5 mm) are directed to a wet double-deck secondary screen with 37.5 mm and 12.5 mm apertures. The undersize fraction (-12.5 mm) from the primary screen is transferred directly to the final product stockpile for blending.

The secondary screen is a wet screen used to wash the feed material, removing surface contaminants, dust, and dirt that could interfere with X-ray Fluorescence (XRF) analysis and may result in inaccurate sorting. The coarse (-100 +37.5 mm) and fine (-37.5 +12.5 mm) products from the wet screen are transferred to 10-tonne coarse and fine sorter feed bins to ensure a consistent feed to the sorters. Each size fraction is processed separately by a sorter calibrated for its specific feed size.

Based on the test results, it was established that the XRF sorting technology would be better suitable for this project. Sorter accepts, or concentrates, are conveyed to the concentrate stockpile, while rejects are sent to a reject stockpile. The sorter is also designed with the capability to produce a middling product, if required.

The wet screen undersize, primarily composed of water and fine dirt, is sent to a belt filter for solid-liquid separation. The filtered cake is transferred to either the final product stockpile or the waste stockpile, depending on its grade, while the filtrate is recycled back to the wet screen as spray water.

The sorter concentrate is blended with the fines (-12.5 mm) from the dry triple-deck primary screen, loaded into bulk trucks, and transported to the port at Stewart, BC, for overseas shipment to the smelter facility.

The sorter waste stockpile is ultimately returned to the pit as fill material, forming part of the final reclamation strategy.

17.4.1 Reagents and Consumables

An anti-scale chemical will be added as needed to minimize scale build-up in the recycle water lines. The reagent will be supplied in liquid form and metered directly into the intake of the water pumps. No other reagents are required for the current flowsheet.

Major consumables will include screen media/decks and jaw crusher liners, which are critical components that require periodic replacement for the operation. Additional consumables include belt filter cloths and laboratory supplies. Maintenance spares for crushing, screening, conveying and assaying equipment will also be provisioned.

17.4.2 Assay and Metallurgical Laboratory

An assay laboratory in the processing plant building provides all the routine assays for the mine, the processing plant, and the environmental and geological departments. The main instruments will include:

- An inductively coupled plasma mass spectrometer (ICP-MS)
- A Leco device
- Fire Assay related devices and instruments

A metallurgical laboratory will be established to undertake all necessary tests to monitor metallurgical performance and, more importantly, to improve process flowsheet and efficiency. The laboratory will be equipped with the following:

- Laboratory jaw and cone crushers
- Ro-Tap® sieve shaker and test sieves
- Ring and puck pulveriser
- Oven-style moisture determination equipment
- Laser particle size analyzer
- pH meters

- Convection oven
- Weighing devices
- Fume hoods with extraction fans
- Dust collection system
- Sample preparation equipment, including laboratory glassware and reagents

Appropriate samplers will be available for routine bulk sampling and plant surveys for process control and metallurgical accounting. The LECO machine is used to conduct Neutralization Potential testing for road materials and environmental monitoring.

17.4.3 Air and Water Services

The compressed air system for the plant is detailed in Section 18.12, while the water supply system is outlined in Section 18.9.

Sorter waste is ejected using pressurized air, making the sorter one of the most air-demanding units in the plant. The required air volume and pressure for the sorters vary depending on the particle size being processed and the proportion of waste rejected. Compressed air for the two ore-sorters will be supplied by a dedicated 15 kW, 480V, 3-phase, 60 Hz air compressor with an air delivery range of 27.5 to 95 CFM at 125 PSI. The system includes a compressor, a dryer, and individual 500L vertical air receivers for each sorter.

Water required for the wet screen is recycled using a belt filter, with occasional freshwater added as needed. Aside from regular clean-up activities, there is no significant process water consumption based on the current flowsheet.

17.4.4 Process Control and Instrumentation

The plant control system will feature a distributed control system (DCS) integrated with personal computer (PC)-based operator control stations (OCSs) located in the plant control room. This control room will be staffed by trained operations personnel.

The DCS will manage equipment and process interlocks, control functions, alarms, trending, event logging, and report generation. Input/output (I/O) cabinets for the DCS will be housed in electrical rooms and connected through a plant-wide fiber-optic network. Programmable logic controllers (PLCs) and other third-party control systems—such as those included with mechanical packages like ore-sorters—will interface with the DCS via Ethernet network connections.

17.4.5 Staffing

Personnel requirements are determined based on operational needs, shift coverage, equipment attendance, safety protocols, training, and maintenance demands. The average annual staffing includes 5 operating staff, 14 operating labourers, and 9 maintenance personnel, totaling 28 individuals on the payroll. Staffing is structured around crews working a single 12-hour shift each day.

18.0 PROJECT INFRASTRUCTURE

18.1 Overview

The Project will require the development of a number of infrastructure items. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and for efficient and convenient operation of the mine equipment fleet. Buildings will be equipped with high pitched roofs for efficient snow management.

The Project surface infrastructure will include the following:

- The existing internal access road network with future upgrades, realignment and new segments to facilitate efficient logistics on Project site
- An ore sorting plant with stockpiles and product loadout
- A modular 160-person accommodation camp with arctic corridors
- A surface mobile equipment shop and warehouse with mine dry and offices
- A cold storage warehouse
- Detonator and explosive storage magazines
- A surface maintenance laydown and storage area
- A power plant with a 13.8 kV power distribution system
- Water supply and distribution
- Fuel storage and distribution
- A site communication system
- Waste management facilities
- A sewage treatment facility
- Surface water management structures
- Mine and sorted waste rock storage facility
- Plant feed storage
- Avalanche mitigation structures

For infrastructure and facilities related to underground mining, refer to Section 16.

For the ore sorting plant, including the comminution and ore sorting equipment, refer to Section 17.

18.2 Site General Arrangement

The Project site general arrangement is presented in Figure 18-1. Details of the mineral resource model and the mine design are described in Section 14.0 and 16.0, respectively.

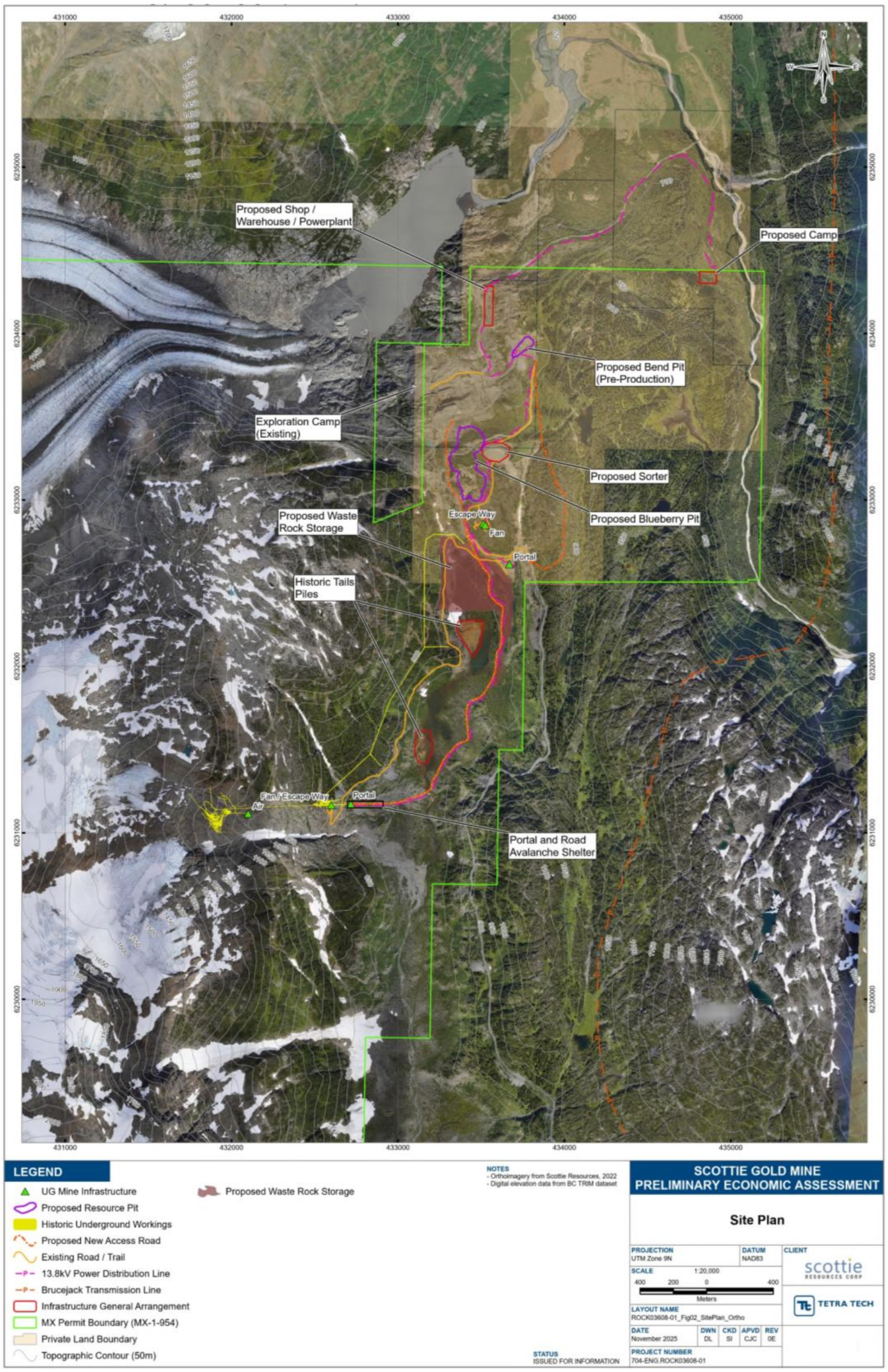


Figure 18-1: Site General Arrangement (Tetra Tech, 2025)

18.3 Access Roads

The Project site is currently accessible via Provincial Highway 37A from Stewart, BC to Hyder, Alaska then by the existing Granduc Road which terminates near an existing mining camp northeast of the Project site (Figure 18 1). The Granduc Road is an all-weather, gravel road that is currently in use by sc ot Resources’ Premier Mine located nearby. The Granduc Road was also used as the principal site access road by mine operations in the past, including the Granduc Mine and the Scottie Gold Mine. Two sections of the Granduc Road will require realignment in the Blueberry Open Pit and the Bend Open Pit areas to accommodate mining activities (Figure 18 1).

The existing internal access road network at the Project is presented in Figure 18 1. These internal access roads will require upgrading to support construction, operation and maintenance activities for the Project where required. Two new haulage roads connecting the Blueberry and Scottie underground portals are proposed (Figure 18 1). The existing access roads will be utilized as much as practically possible to minimize the environmental footprint and construction costs. Avalanche mitigation structures will be installed where necessary, complimented by active avalanche monitoring and controls during winter.

Access Controls

The site access roads connected to Granduc Road, will have a modular, motorized access control gate to prevent accidental entry into the Project site, for safety and security (Figure 18-2). The access road to the detonator and explosive storage will have another modular access control gate for restricted access. The access control gates will be monitored and can be operated by RFID key cards or remotely by the site security team.

Figure 18-2: A Typical Motorized Access Control Gate (Tetra Tech, 2025)

18.4 Accommodation Camp

On-site camp accommodations for operation personnel will be provided during the operation phase. The current design includes:

- Modular dormitory units to provide 160 single-occupancy rooms
- A kitchen and cafeteria

- A recreation room
- An administration office
- Arctic corridors
- Potable water disinfection
- Sewage treatment
- Incinerator
- High-pitched roofs

The accommodation camp will be erected and commissioned during the early stage of project development to provide 160 single-occupancy rooms during construction and operation. Each room will be equipped with a washroom, a shower, plumbing fixtures, furniture including a bed, a closet and a desk, lighting, fire detection, thermostat-controlled heaters, power outlets, and Wi-Fi. Rooms will be assigned on a check-in/check-out basis. Common areas in each dormitory complex will be equipped laundry rooms, fire detection and protection equipment and mud rooms with boot racks. Rooms and common areas will be cleaned daily by the housekeeping team.

The mess hall area will be sized to accommodate 80-persons and provide hot foods during breakfast, lunch and dinner hours. Pre-packed meals will be available for personnel to take away and consume mid-shift. The kitchen will be of processional grade and operated by meal preparation staff who are food safe certified. Catering and housekeeping services will be provided by a licensed contractor. Fire detection and protection equipment will be installed where required.

The camp modules will be interconnected with arctic corridors and equipped with high-pitch snow roofs, similar to the Eskay Creek Mine Camp, for effective snow clearing during winter (Figure 18-3). The envisioned accommodation camp dormitory interior and floor plan are presented in Figure 18-4. Daily shuttle bus services will be provided for transporting the workers between the camp and the work areas.

The potable water supply system near the camp will consist of water filters and a chlorination and UV disinfection system. The treated potable water quality will meet or exceed drinking water standards set forth by the relevant regulator. The potable water supply system will be subject to routine inspections and sampling. A water bottling system inside the kitchen will provide drinkable water to offices outside the camp. The camp will include a modular sewage treatment facility (see Section 18.13). An incinerator near the kitchen will provide an effective means for disposing putrescible waste while minimizing wildlife attraction. An LNG tank near the camp will provide the energy storage and supply for heating and cooking.

Figure 18-3: Accommodation Camp Modules with High Pitched Roofs (Skeena Gold, 2025)

Figure 18-4: Accommodation Camp Dormitory Interior and Floor Plan (Tetra Tech, 2025)

18.5 Warehouse/Shop/Mine Dry & Offices

The warehouse/shop/mine dry/office building will be a pre-engineered tension fabric building to accommodate maintenance and repair, warehouse storage, offices, dry areas, and general storage facilities. There are successful examples of pre-engineered tension fabric buildings used for mine operations in sub-arctic regions, such as the Meadowbank Mine in Nunavut (Figure 18-5). The building structure is supported by standard cargo containers at the base.

The material and install costs of pre-engineered tension fabric buildings are considerably less than traditional steel buildings.

Figure 18-5: Meadowbank Mine Equipment Maintenance Complex (Agnico-Eagle Mines, 2025)

This building will comprise:

- Two (2) heavy equipment maintenance bays
- Two (2) light vehicle repair bays
- A 20-ton overhead service crane
- A machine shop
- A welding shop
- An electrical shop
- A storage warehouse with an upper level mezzanine area for offices
- A mine dry area including 120 lockers, washrooms and showers
- A first aid room
- Emergency vehicle storage
- Building services such as power, water, drainage, a septic tank, HVAC, a mechanical service room and an electrical room.

18.6 Storage Warehouses

The 400 m^2 storage warehouse will be located adjacent to the warehouse / truck shop / mine dry building. It will be a pre-engineered tension fabric building with metal roll-up doors. This building will store equipment, parts and supplies sensitive to ambient temperature or environment. Equipment, parts and supplies that are not sensitive to temperature will be storage in seacan containers in the laydown area near the shop.

Another 400 m^2 storage warehouse will be erected near the Blueberry underground portal (Figure 18-1) for UG mine supplies. Similarly, it will be a pre-engineered tension fabric building with metal roll-up doors.

Scottie Resources currently owns a warehouse in Stewart, BC, which can supplement additional storage space to support Project construction and operation.

18.7 Waste Rock Storage

A preliminary waste stockpile was designed to contain the total waste volume from both the open pit and underground mining areas (Scottie and Blueberry underground), Figure 18-1. The total storage capacity is roughly 4 Mm^3, while the total waste expected from the open pit and underground combined is approximately 3.6 Mm^3. Waste will be required to backfill the Scottie historical underground workings. There may also be an opportunity to backfill waste into the Blueberry pits with underground waste from Blueberry.

18.8 Power Supply and Distribution

Power supply at Project site will be provided by a power generation plant on site that will consist of five 1.4 MW natural gas-powered generator sets; four continuous and one standby units (N+1). Each power generator and system controls will be modularized in a shipping container, fabricated, tested and pre-commissioned at the factory before shipping to the Project site to minimize the field installation and commissioning labour hours. Cable and wire connections will be of the “plug and play” type to minimize installation hours on site. The plant is designed such that additional modular units may be easily added in the future.

The power generators will be of modular design to significantly reduce construction costs and site labour requirements and skid-mounted to allow fast deployment and placement on a compacted gravel pad without a concrete foundation. The generators will be equipped with day tanks and drip trays to contain and prevent escapement of any fugitive fluids. The generator enclosures and exhaust will have sound attenuation material to reduce the noise level. A pre-engineered tension fabric building with a high-pitched roof will house the generators for further noise attenuation and efficient snow clearing in winter.

Each power unit will be arranged with interconnecting walkways and fire doors. Programmable logic controllers will be used for fully automatic operation of the power plant, including alarm annunciation, data logging and remote fault reporting. The control system will be fully automatic to start and stop individual generator sets to best match the prevailing load. All modules will have fire detection panels and inert gas fire suppression systems. The switchgear and control module will also include the plant feeder breakers. An emergency diesel generator will be provided at the power plant to provide power for a “black start” of the plant.

The planned natural gas power plant design is based on conventional simple cycle, four stroke, turbocharged, aftercooled, electronically injected, high speed, natural gas engines, with total waste heat recovery equipment for providing heating to the buildings nearby. The engines will drive 13.8 kV, 0.8 power factor, high efficiency synchronous generators. The natural gas generator plant design has been selected on the basis of the relatively low capital cost of installing modular units while the ongoing operating costs are reasonable for natural gas fuel units based on the relatively high thermal efficiency of the electronically controlled engines. The selected power generation option is cost effective and practical for this remote site and offers a balance between these considerations and those of environmental impact, socio-economic impact and amenability to reclamation.

The natural gas power generation was selected based on the conceptual level power supply trade-off study conducted by Tetra Tech. As per meeting between Tetra Tech and BC Hydro on June 24, 2025, Tetra Tech was informed that the 138 kV power transmission line about 1.5 km from the Project site connecting the LNT substation and the Brucejack Mine (Figure 18-1), currently does not have sufficient spare capacity to supply the power for a new large-scale industrial customer such as the Scottie Gold Mine. However, certain future events, such as power infrastructure upgrades, business acquisitions and/or power supply sub-agreements by other mining accounts, may create an opportunity for the Project to acquire power from this existing 138 kV power transmission line. The opportunity is discussed in Section 26 of this Technical Report.

Site Power Distribution

The site power distribution general arrangement is present in Figure 18-1. Power will be distributed from the power plant to all end users via the 13.8 kV distribution system. Prefabricated electrical Houses (E-houses) will be used for housing the transformers, switch gears, instrumentations, controls, cooling fans, lightning protection and grounding. E-houses will step down 13.8 kV to 600 V for equipment and 240/120 V for lighting and general usage. Large equipment, such as UG mine ventilation fans, will run on 600 V and will require 13.8 kV to 600 V step down transformers. The sorting plant will require a 13.8 kV to 480 V transformer, as this equipment is provided in 480 V from the manufacturers.

All E-houses will be modularized in shipping containers, fabricated, tested and pre-commissioned at the factory before shipping to the Project site to minimize the field installation and commissioning labour hours. Cable and wire connections will be of the “plug and play” type to minimize installation hours on site. Modularized E-houses can be positioned on compacted gravel pads without concrete foundations.

Emergency Power Distribution

There will be two 500 kW emergency generators at the plant site, retained and refurbished after construction use. The emergency generators will be distributed around the site to pick up critical loads via transfer switches. The generators will be located near the critical loads. Critical loads will be camp, communications and minimal mine loads.

18.9 Water Supply and Distribution

Supply of fresh water is assumed to be drawn from groundwater sources, via water wells located within the Project site, one which will be operational and one standby. A freshwater tank will be located at the camp complex and will be distributed by pumps for potable water feed and for general use.

Fresh water for potable water use will be treated via multimedia filtration and disinfection by chlorination and ultraviolet rays. The potable water treatment module will be part of the accommodation camp facility and sized for approximately 160 personnel on site. Treated water will be distributed in a piping ring to serve all potable water users in the camp complexes. The camp will have a bottling facility for the distribution of drinking water to other Project facilities. Refer to Section 18.4 for more details.

The clean service/fire water tank located near the maintenance shop will have a reserve in the lower portion of the water tank and will be drawn from below the primary water nozzles by the fire water pumps. Fire water pump skids complete with a diesel-driven fire pump, jockey pump and controls will be installed. Dedicated fire mains complete with hydrants will be provided at the major site buildings. Fire extinguishers will also be provided throughout the facilities. Sprinkler systems will be installed in the warehouse, the main office and the shop. Fire alarm systems at the site facilities will report to the emergency response/first aid unit at mine site which will be monitored 24 h/d.

The feasibility of using groundwater as a water supply source should be further reviewed during future phases of the Project with respect to sustainability of yield, water quality, as well as environmental and permitting considerations.

18.10 Diesel Fuel Storage Farms and Distribution

Certified modular double-walled diesel fuel tanks (also known as "IS tanks") will be delivered to Project site by the fuel supplier and serve as mobile fuel storage tanks on site. These tanks will be laid down in a HDPE-lined and contained fuel farm areas near the power plant and the portals. A modular fuel dispensing unit equipped with fuel guns will provide a means for equipment refueling.

The double-walled modular tanks will have visual vacuum gauges to indicate any possible leakage between the walls. The leak and gas vapour detection Instrumentations in the fuel farm area will provide audible alerts locally and wireless alerts remotely to the site operation center. Any fuel leakage will be contained by the HDPE-liner and containment berms in the fuel storage area.

18.11 LNG Fuel Storage and Regasification

The power plant will include LNG storage and regasification equipment mounted on the graded surface, which will be used to receive, store, and supply LNG to the power plant. The area consists of an LNG offloading pump station to transfer LNG from shipping tankers to insulated storage tanks, a vapourizer for turning LNG to gaseous form, a waste heat vapourizer, water/glycol pumps for building heating, a control room, and pipeline infrastructure for transporting LNG within site. The storage area will house sufficient LNG storage tanks to provide about one week of LNG fuel storage capacity.

A modular LNG fuel gas tank will be installed near the camp for providing heating and cooking fuel.

18.12 Mine Plant and Instrument Compressed Air

Compressed air service for plant and instruments will be provided by local air compressors, receivers and dryers located inside the plant or building, where required. A site wide compressed air system would not be practical, due to the distances between buildings.

18.13 Communications

The telecommunications design for the Project will incorporate proven, dependable, and state-of-the-art systems to ensure that personnel at the mine site will have adequate data, voice, and other communications channels available.

The telecommunications system will be supplied as a design-build package. The base system will be installed during the construction period then expanded to encompass the operation. The design will include:

- A Voice over Internet Protocol (VoIP) telephone system
- Satellite communications for voice and data
- Ethernet cabling for site infrastructure
- Wireless Internet access
- Two-way radio communications at site
- Satellite TV

A main telecommunication central equipment office will consist of a pre-manufactured trailer in which the main communications contractor will install and test all the main sub-systems for the facilities, prior to shipment. The trailer will form the first block in a system that must support the construction needs of the Project first, and the operating needs of the Project following construction.

Spare parts for critical and main components will be provided to ensure maximum reliability, and minimum down time. A variety of communications media (copper and wireless during the construction phase and fibre optic during the operating phase) will be incorporated in the overall design.

The requirements for communications, particularly satellite bandwidth, are a function of the voice and data requirements of the active participants in the Project.

Technologies and services to be provided include the following:

Construction Phase:

- Local VoIP wireless network
- Satellite link for voice, data and video services
- PC Local/Wide Area Network (LAN/WAN)
- Trunked mobile radio system

- Internet service
- Private telephone system for voice and fax
- Video conferencing to minimize travel during design and construction
- Ground-go-air communications system (VHF radio)
- Cable television on independent satellite system

Operations Phase (includes selected services above):

- Process monitoring and control for efficient operation and maintenance
- Fibre optic cabling for plant wide communications
- Security access control
- Remote equipment monitoring capacity from corporate offices in Stewart, BC and Vancouver, BC.
- Internet services for workers

A satellite based back up communication system, such as Starlink™, will provide redundancy.

Communications in the underground mine will be supported by a leaky feeder system.

18.14 Waste Management

Domestic solid waste generated over the course of the Project is proposed to be incinerated on site, as per typical mining operations in northern BC. Domestic waste generated during the operations phase is approximated to be 0.2 t/d based on an expected on-site workforce of 160.

Hazardous and non-hazardous industrial solid wastes such as tires, waste lumber, scrap metals, plastics, and electrical wastes will be collected and sorted prior to off-site removal by a qualified local contractor for recycling and/or disposed at an appropriate facility. Hazardous liquid wastes such as laboratory chemical wastes and paints will be collected and temporarily stored on site, with final disposal off site by a licensed contractor. No long-term, on-site waste storage areas have been included as part of the Project.

18.15 Sewage Treatment

The accommodation camp will include a modular sewage treatment plant which will treat the organic wastewater generated in the camp, prior to discharging to the surroundings. The sewage treatment plant will be designed to meet the relevant federal and provincial regulations and effluent quality guidelines.

Sewage from the offices and mine dry will be stored in local underground septic tanks. The sewage will be regularly removed by a vacuum truck and transported to the sewage treatment facility at the accommodation camp for treatment.

18.16 Surface Water Management

A site wide water balance will be prepared during the next phase of the Project study to identify the water quality and quantity of all water sources on site and from the surroundings, which will guide the design of all surface water management infrastructure and facilities on site.

For the PEA, cost allowances were included in the cost estimate for:

- Mine dewatering
- Uncontacted water diversion
- Collection, storage, pumping, and treatment of contact water on site, including the raw plant feed wash water from the ore sorting plant

Sediment Control Plan

Normal industry practices (such as silt fences, containment ponds, etc.) will be used during the construction of any of the facilities.

Non-contact storm water from above the disturbed areas/facilities will be controlled and diverted before it can become contaminated (i.e. become contact water), thus reducing the sizing of sediment control structures.

All contact water at the plant site will be directed to the sediment control pond located adjacent to each infrastructure pad. Solids will be allowed to settle out and the water will be removed for site water use, or pumped to the portable water treatment modules or released, based on the water quality. The contact water will be treated prior to releasing, according to applicable regulations and water quality standards.

18.17 Ore Sorting Plant Storage Stockpiles

The ore sorting plant stockpiles will be located in the ore sorting plant. Refer to Section 17.0 for details.

18.18 Assaying Services

A modular, trailer-type assay laboratory will be installed within the sorting plant to provide the necessary assaying services to support the operation.

18.19 Avalanche Monitoring and Mitigation

Some areas of the Project are subject to avalanche hazards, as shown in Figure 18-6; these have been identified as subject to the risk of avalanches. Where possible, potential avalanche areas will be avoided when locating facilities. However, in any areas where this is not feasible, power lines and pipelines will be buried for physical protection. Avalanche mitigation structures are included in the Project design. Snow fences will be the primary structures for avalanche mitigation. A proactive avalanche monitoring and management plan will be put in place during the construction period and will be continued throughout the operational period of the mine. Computer-based analytical tools will provide forewarnings based on long and mid-range weather forecasts and information, advisories or warnings issued by organizations such as Avalanche Canada® or US National Weather Service.

Remote avalanche control systems will likely be the most practical solution to reducing the chances of a high magnitude avalanche. In that situation, avalanche areas will be monitored and controlled blasting will be used to minimize avalanche effects; if determined to be the solution, a remote system will be available 24/7 during all weather conditions. Avalanche awareness and control will be an essential safety activity, in which all project personnel will be instructed. The site avalanche monitoring and mitigation system is presented in Figure 18-7.

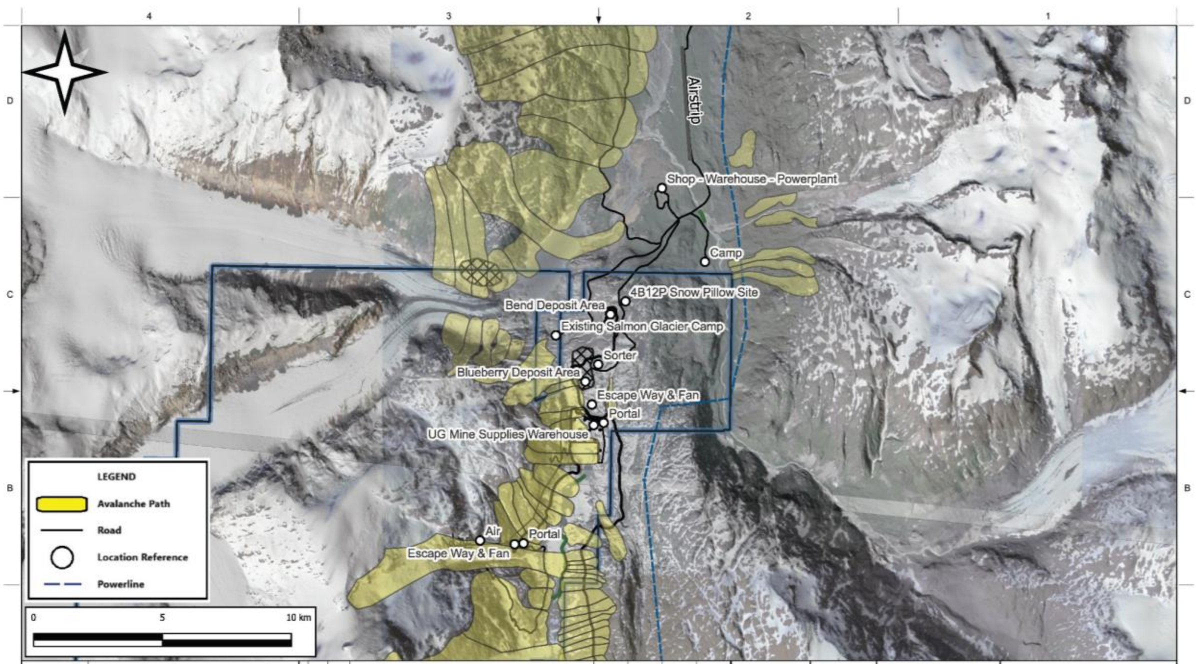


Figure 18-6: Site Avalanche Assessment Map

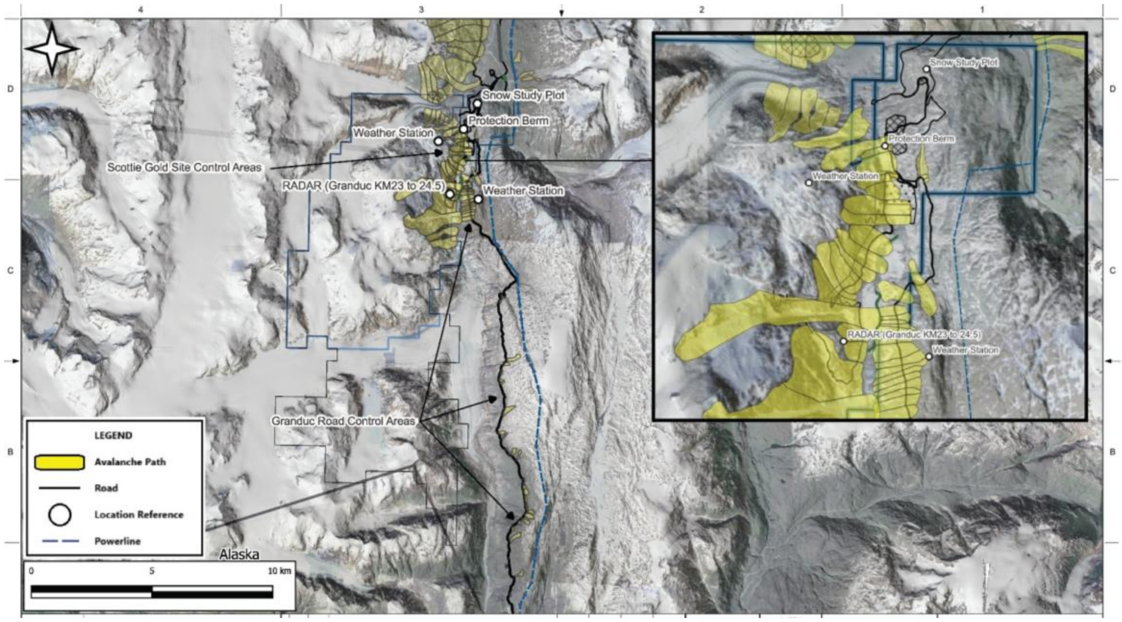


Figure 18-7: Site Avalanche Monitoring and Mitigation System Map

18.20 Off-Site Infrastructure: Port Facilities

Two existing ice-free port facilities in Stewart BC, the Stewart Bulk Terminals and the Stewart World Port, are capable of providing material storage, handling and shipment by ocean-going vessels. The Stewart Bulk Terminals is capable of handling 50,000 dwt ships and offers roll-on/roll-off services to barges. The deep sea wharf at the Stewart World Port is capable of berthing Handymax and Panamax vessels, with a break bulk wharf capable of accepting roll-on/roll-off and break-bulk cargo. Both ports are equipped with equipment capable of rotating dump truck for material off-load and material handling systems to transport materials and load vessels. The intent for the project is to use bulk shipping methods.

19.0 MARKET STUDIES AND CONTRACTS

The product to be produced by the Project is crushed gold-bearing mineralized material, concentrated by the ore sorting process. The product production schedule and product characteristics are presented in Section 17.

From a logistics perspective, given the location of the Project and its proximity to the ice-free seaports in Stewart, BC, gold concentrates produced from this location will be competitive selling into the Asian market.

The product produced on Project site will be transported by highway licensed trucks in bulk form from the Project site to the designated seaport at Stewart, BC, and shipped by ocean-going vessels to the designated smelter/buyer located in East Asia.

The marketing and shipping related costs, such as gold payables, treatment charges, refining charges, insurance, transportation and product handling, are presented in Section 22.

Scottie Resources has entered into a binding agreement with Ocean Partners to purchase all DSO production from the existing resource at the project. The agreement outlines the payment terms for all of this production.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Setting

The property is located in northwestern BC as illustrated in Figure 4-1, within the boundary ranges of the Coast Mountains and within the Nass Area under the governance of the Nisg̱a’a Lisims Government (NLG) as defined in the Nisg̱a’a inal gre ement (N), and within the Consultative Area Zones of the Tsetsaut Skii Km Lax Ha.

The elevation on the property ranges from a low of 700 m above sea level (asl) to a high of 2126 m asl, found at the peak of Summit Mountain. The eastern portion of the property is hosted to slopes with lower elevation which are sparsely vegetated with spruce trees, alder trees, and blueberry bushes, where the western portion of the property has a local topography of alpine and sub-alpine.

Field work on the property is typically carried out from late June to October; however, the Granduc Mine Road could be kept open throughout the year with the use of an adequate avalanche control program and a snow removal team. The well-maintained Granduc Mine Road passes through the claim boundaries, which provides access to the northeastern areas of the Property. Helicopter support is used to provide access to the remaining portion of the property, with the closest airbase found in the town of Stewart, BC.

The climate normals from 2011 to 2020 were obtained from ClimateBC, a Microsoft Windows-based computer program that generates scale-free climate data for specific locations and for gridded climate data at any spatial resolution. The program is developed and maintained by the Centre of Forest Genetic Conservation, Faculty of Forestry, the University of British Columbia (UBC) and sponsored by the BC Ministry of Forests. The obtained monthly data are presented in Table 20-1.

Table 20-1: Climate Normals (2011 to 2020) for Average Air Temperature and Total Precipitation Generated by ClimateBC for the Project Location

Period	Average Air Temperature	Total Precipitation (mm)
January	-5.4	301
February	-4.8	204
March	-1.0	129
April	3.2	109
May	8.2	81
June	10.7	74
July	12.3	78
August	12.4	157

Period	Average Air Temperature	Total Precipitation (mm)
September	8.5	198
October	2.6	289
November	-2.7	297
December	-5.8	260

20.2 Environmental Baseline Program

Environmental baseline studies were initiated in Q3 2025 to historical data collected since 1983 and on-going data collection started in 2023. The environmental baseline studies include archaeological, aquatics, terrestrial, hydrometric, meteorological, hydrogeological, and geochemical data collection. Data collection will continue for at least two years, and the results will support regulatory authorization applications.

20.3 Project Permitting Requirements

Scottie currently possesses the necessary permits to conduct exploration and drilling activities on site.

Scottie holds Mines Act Permit MX-1-954 for exploration activities including drilling, trail construction, camp construction, and operation and reclamation of the prescribed activities that was most recently amended in July 2025, to include a 10,000 tonne bulk sample in the permitted activities.

In July 2025, Scottie received all required permits from the Government of BC to proceed with a 10,000-tonne surface bulk sampling program, which allows Scottie to excavate up to 10,000 tonnes of rock (including mineralized material and waste) and take samples from the road-accessible outcropping Bend Vein located on the north end of the project. Scottie expects to produce up to 4,000 tonnes of mineralized rock from the program.

Scottie Gold Mine is an existing mine in Care and Maintenance under permit M-139, and does not require review under the British Columbia Environmental Assessment Act or under the federal Impact Assessment Act. However, it will require multiple permits and amendments to existing permits, including a provincial Mines Act permit and Environmental Management Act permit.

All other potentially required provincial and federal authorizations required for the Project are summarized in Table 20-2.

A list of key regulatory instruments that may be applicable to the project include:

- Environmental Assessment Act (BC)
- Environmental Management Act (BC)
- Forest Act (BC)
- Forest and Range Practices Act (BC)

- Forest Practices Code of British Columbia Act (BC)
- Health Act (BC)
- Health, Safety and Reclamation Code for Mines in British Columbia (BC)
- Industrial Roads Act (BC)
- Land Act (BC)
- Mineral Tenure Act (BC)
- Mines Act (BC)
- Mining Right of Way Act (BC)
- Motor Vehicle Act (BC)
- Nisg̲a’a in al greement ct (BC)
- Safety Standards Act (BC)
- Transportation Act (BC)
- Water Sustainability Act (BC)
- Wildlife Act (BC)
- Canadian Environmental Protection Act (BC)
- Canada Transportation Act (BC)
- Transportation of Dangerous Goods Act (BC)
- Canada Explosives Act (BC)
- Navigations Protection Act (Canada)
- Fisheries Act (Canada)

Table 20-2: Preliminary List of Potential Provincial and Federal Authorizations Required for the Project

Permit/Approval	Provincial Statute	Responsible Agency	Project Activity
Waste Discharge Permit and Waste Storage Approval	*Environmental Management Act*	Ministry of Environment and Parks	Permitting system to enable authorized discharge of effluent to water, storage/treatment of wastes, disposal of solid waste to land, and discharge of emissions to the atmosphere.

Permit/Approval	Provincial Statute	Responsible Agency	Project Activity
Heritage Conservation Act s. 14 Heritage Inspection Permit or Heritage Investigation Permit; s. 12 [Site] Alteration Permit	*Heritage Conservation Act*	Ministry of Forests: Archaeology Branch	Heritage inspection, investigation, or site alteration of lands potentially affected by the project.
Heritage Conservation Act Concurrence letters	*Heritage Conservation Act*	Ministry of Forests: Archaeology Branch	Assessment under the *Heritage Conservation Act* must be completed prior to the commencement of ground disturbing activities.
Wildlife Act Permit	*Wildlife Act*	ENV: Environmental Stewardship Division	Wildlife salvages and surveys of wildlife and their habitat. Bird nest removal or relocation.
Construction Permit for a Potable Water Well	*Drinking Water Protection Act*	BC Ministry of Health, Northern Health Authority	Groundwater well for domestic water use.
Water System Construction Permit	*Drinking Water Protection Act*	BC Ministry of Health, Northern Health Authority	Construction of a potable water system.
Drinking Water System Operations Permit	*Drinking Water Protection Act*	BC Ministry of Health, Northern Health Authority	Operation of a potable water system.
Short Term Use of Water Permit	*Water Sustainability Act*	Ministry of Environment and Parks: Water Stewardship Branch	Short-term use of water from freshwater streams and lakes for construction purposes.
Water Sustainability Act Approval	*Water Sustainability Act* and BC Dam Safety Regulation	Ministry of Forests	For changes in and about a stream including diversions, storage, and use of water, including management of nuisance water from mining operations.
Water License	*Water Sustainability Act*	Ministry of Forests	For construction and operation of Project activities requiring diversion of surface waters or groundwater sources for potable or process water.

Permit/Approval	Provincial Statute	Responsible Agency	Project Activity
Licenses to Cut and Special Use Permit	*Forest Act*, Part 3, Section 8.2 License to Cut Regulation Provincial Forest Use Regulation	Ministry of Forests: Forest Tenures Branch	License to Cut Permit to harvest in a specific area over a relatively short time period. Special Use Permit to gain nonexclusive authority to use Crown Land within Provincial Forest, if in accordance with Provincial Forest Use Regulation (annual rent and taxes apply) for the construction or maintenance of a road, bridge, or drainage structure, weather station, weight scales, or quarries used for road construction or maintenance.
Industrial Access Permit	*Transportation Act*	Ministry of Transportation and Infrastructure	Required for any new roads that join onto public roads controlled by the Ministry of Transportation.
Permit for regulated activities	*Public Health Act*	Ministry of Health	Regulated activities may, if prescribed standards are not met, endanger health or cause injury or illness, or are not regulated under an enactment (or if regulated do not sufficiently prevent, mitigate or respond to the risk to health or risk of injury or illness). Such activities could be providing potable water, processing wastewater, or managing septic systems.
Hazardous Waste Generator Registration	*Environmental Management Act* Hazardous Waste Regulation	Ministry of Environment and Parks	A registration process for the owner of a waste (e.g., property owner) identified as being hazardous to detail the steps taken to store hazardous waste at the generation location.
Sewage Registration	*Environmental Management Act* Municipal Sewage Regulation	Ministry of Environment and Parks	Registration identifying specific information regarding the sewage discharge activities.
Food Service Permits	*Health Act*	Provincial Health Services Authority	To operate a kitchen in a mining camp.
Authorization under Paragraphs 34.4(2)(b) and 35(2)(b)	*Fisheries Act*	Fisheries and Oceans Canada (DFO)	Conducting work or activities that result in the death of fish or that result in the harmful alteration, disruption or destruction of fish habitat.
Migratory Birds Convention Act Authorization	*Migratory Birds Convention Act* and Migratory Bird Sanctuary Regulations	Environment and Climate Change Canada (ECCC)	Deposit of substances harmful to migratory birds or vegetation clearing during the migratory bird nesting season as outlined by ECCC for the Project area, Zone A2, early

Permit/Approval	Provincial Statute	Responsible Agency	Project Activity
			April to mid-August (ECCC, 2018).
Species at Risk Act Permit	*Species at Risk Act*	ECCC, DFO, Parks Canada	Authorizes activities that will affect a listed wildlife species, any part of its critical habitat, or the residences of its individuals.
Explosive Licenses and Permits; Ammonium nitrate storage Approval	*Explosives Act*, and Regulations; Ammonium Nitrate Storage Facilities Regulations	Natural Resources Canada; Transport Canada	Explosive License required for factories and magazines. Explosive Permit required for vehicles used for the transportation of explosives.
Transportation of Dangerous Goods Permits	*Transportation of Dangerous Goods Act*	Transport Canada	Related to the classification, documentation, marking, means of containment, required training, emergency response, accidental release, protective measures and permits required for the transportation of dangerous goods by road, rail or air.
Storage Tank System Registration and Allied Petroleum Products Regulations	*Canadian Environmental Protection Act Storage Tank Systems for Petroleum Products;* Storage Tank Systems for Petroleum Products and Allied Petroleum Products Regulations	ECCC	For the storage of fuels aboveground and underground used for storage of petroleum and allied petroleum products with a capacity greater than 2,500 L.

20.4 Indigenous Nations, Consultation, and Community Engagement

Indigenous Nations and Rights Holders

The Project is located within the area covered by the Nisga'a Nation N , Canada's first modern treaty. The Nisga'a Nation has rights to environmental participation, land use, and resource consultation in accordance with the NFA. Scottie has initiated engagement with NLG and the Nisga'a Growth Corporation to incorporate participation in the development of the project, support capacity development for the Nation, and employ Nisga'a Nation members and businesses.

The project is also located within the Tsetsaut Skii km Lax Ha (TSKLH) Consultative Area Zones. Previously, TSKLH was represented by the Gitxsan Chiefs Office, but since 2005, TSKLH has asserted its Tsetsaut identity and heritage. The Gitxsan Treaty Society or Gitxsan Hereditary Chiefs Office, acknowledging TSKLH governance, has directed proponents to engage directly with TSKLH[1].

Alaska tribes – specifically the Metlakatla Indian Community and the Southeast Alaska Indigenous Transboundary Commission (SEITC; comprised of 7 Alaskan Tribes) – may also be engaged due to the transboundary transportation of product through Alaska.

Local Government and Public Engagement

Initial outreach has occurred with the District of Stewart and various overlapping interest holders. Key interests include:

- Local economic development and employment opportunities
- Traffic and road use impacts
- Recreational users, particularly snow mobile users in the area
- Heli skiing

As the project progresses and more information is gleaned, future engagement will be expanded to include more local stakeholders, land users, and community organizations.

Social Considerations

The project has potential to provide employment and business opportunities in a remote region. Early planning includes local hiring and procurement strategies. No known non-Indigenous land users have raised concerns to date, but this will be confirmed through expanded engagement.

---

[1] Skeena Resources, Eskay Creek Revitalization Project Revised Application for an Environmental Assessment Certificate / Impact Statement, Chapter 7, Tsetsaut Skii Km Lax Ha. Available at: https://www.projects.eao.gov.bc.ca/p/60f078d3332ebd0022a39224/documents

20.5 Conceptual Mine Closure and Reclamation Plan

The Conceptual Closure and Reclamation Plan for the project includes the following activities:

- Decommissioning and removal of the surface facilities from the project site, including all buildings and equipment, such as the processing plant, power plant, camp, maintenance shops, warehouse, storage, tanks, pumps, assay laboratory, power distribution, mine ventilation, avalanche monitoring and mitigation, explosive magazines, and utilities. Concrete slabs and footings will be broken and placed appropriately. Certain equipment packages, such as the water treatment plant and power supply, will be retained on site to provide post-closure water treatment service.
- Collection and removal of contaminated soils.
- Stockpiles and earthworks, including WRSF, building pads, laydown areas and certain internal roads, will be re-contoured for geotechnical stability, covered and capped with appropriate fill, top soiled and revegetated. Engineered cover systems will be applied where PAG material may be present for ARD mitigation.
- Pits and wells will be backfilled with appropriate NAG fill material, top soiled and revegetated.
- Appropriate seepage collection and piping systems will be installed to collect and transport any seepage to the water treatment plant.

The estimated Closure and Reclamation costs are discussed in section 22.2.

21.0 CAPITAL AND OPERATING COSTS

21.1 Summary

The capital and operating costs for the Project have been estimated and are summarized in Table 21-1.

Table 21-1: Summary of Capital and Operating Costs

Description	Total Capital Cost (C$ million)	Average Unit Operating Cost (C$/t product)
Initial Capital Costs	128.6	-
Sustaining Capital for LOM	76.7	-
LOM On-site Operating Costs	-	185.30

All costs are reflected in 2025 Q3 Canadian Dollars unless otherwise specified. The expected accuracy range of the cost estimates is within ±35%. Where applicable, costs in this report have been converted from US Dollars to Canadian Dollars using a currency exchange rate of CAD1.00:USD0.72.

21.2 Capital Cost Estimate

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $128.6 million. A summary breakdown of the initial capital cost is provided in Table 21-2. This total includes all direct costs, indirect costs, wn er’s costs, and contingency.

Table 21-2: Initial Capital Cost Summary

Description	Initial Capital Cost (C$ million)
Direct Costs	
Mining Infrastructure	6.8
Site Preparation	5.0
Sorting Plant	26.2
On-Site Facilities: Camp, Power Plant Laboratory, and Other Facilities	28.5
Surface Mobile Equipment	7.8
Utilities, such as Fresh/Potable Water, Power Distribution and Waste Management	7.3
Water Management and Avalanche Control	3.0
Subtotal – Direct Costs	**84.6**
Indirect Costs	
Project Indirect Costs	23.7
wner’ s Costs	3.4
Contingencies	16.9
Total	**128.6**

21.2.1 Class of Estimate

This Class 5 cost estimate has been prepared in accordance with the standards of AACE International. The expected accuracy of this estimate is within ±35%.

21.2.2 Estimate Base Date and Validity Period

This estimate was prepared with a base date of Q3 2025 and does not include any escalation beyond this date. Vendor quotations used for this PEA estimate were obtained in Q3 2025 and have a validity period of 90 calendar days or less.

21.2.3 Estimate Approach

Currency and Foreign Exchange

The capital cost estimate uses Canadian Dollars as the base currency. Where applicable, quotations received from vendors were converted to Canadian Dollars using a currency exchange rate of CAD1.00:USD0.72. There are no provisions for foreign exchange fluctuations.

Duties and Taxes

Duties and taxes are not included in the estimate.

Measurement System

The International System of Units (SI) is used in this estimate.

Work Breakdown Structure

The estimate is organized according to the following hierarchical work breakdown structure (WBS):

- Level 1 = Major Area
- Level 2 = Area
- Level 3 = Sub-Area.

21.2.4 Elements of Cost

This capital cost estimate consists of the four main parts: direct costs, indirect costs, wner’ s costs, and contingency.

Direct Costs

AACE International defines direct costs as:

...costs of completing work that are directly attributable to its performance and are necessary for its completion. In construction, (it is considered to be) the cost of installed equipment, material, labor and supervision directly or immediately involved in the physical construction of the permanent facility.

Examples of direct costs include processing equipment and permanent buildings.

The total direct cost for the Project is estimated to be $84.6 million.

Indirect Costs

AACE International defines indirect costs as:

...costs not directly attributable to the completion of an activity, which are typically allocated or spread across all activities on a predetermined basis. In construction, (field) indirects are costs which do not become a final part of the installation, but which are required for the orderly completion of the installation and may include, but are not limited to, field administration, direct supervision, capital tools, start-up costs, contractor's fees, insurance, taxes, etc.

The total indirect cost for the Project is estimated to be $23.7 million.

Owner’s Costs

wner’ s costs are costs provided by the wne r to support and execute the Project.

The Project execution strategy, in particular for construction management, involves the Owner working with an engineering, procurement, and construction management (EPCM) organization and supervising the general contractor(s). The wner’ s costs include home office staffing, home office travel, home office general expenses, field staffing, field travel, general field expenses, community relations, and wne r’s contingency.

The total wner’ s cost allowance for the Project is estimated to be $3.4 million.

Contingency

Tetra Tech estimated a contingency for each activity or discipline based on the level of engineering effort as well as experience on past projects.

The total contingency allowance for the Project is $16.9 million.

21.2.5 Responsibility

Tetra Tech is responsible for the development of the overall Capex with inputs from the following consultants and client.

- Tetra Tech – All areas, including direct and indirect costs, except ore-sorting equipment and wner’ s costs
- ABH Engineering Inc. – Ore-sorting Equipment
- Scottie Resources – wner’s costs

21.2.6 Capital Cost Exclusions

The following items have been excluded from this capital cost estimate:

- Working or deferred capital (included in the financial model)
- Financing costs
- Taxes and duties
- Land acquisition
- Currency fluctuations
- Lost time due to severe weather conditions
- Lost time due to force majeure
- Additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares
- Any project sunk costs (studies, exploration programs, etc.)
- Mining operating costs which are included in financial analysis as pre-production operating costs

- Mine reclamation and closure costs (included in the economic analysis. See Section 22.2)
- Escalation costs
- Any other unforeseeable events or anomaly.

21.2.7 Salvage

Since the mine life is shorter than other comparable projects and typical equipment life, it is expected that most of the processing, power equipment and certain facilities will be salvageable by the end of the seven year mine life. The estimated salvage value is $12.9 million consisting mostly of modular equipment and facilities which can be salvaged upon the end of mine life.

21.3 Operating Cost Estimate

21.3.1 Summary

On average, the LOM on-site operating costs for the Project were estimated to be $185.38/t processed. The operating costs are defined as the direct operating costs including mining, processing, site servicing, and G&A costs, including related freight costs. Table 21-3 and Figure 21-1 show the cost breakdown for various areas.

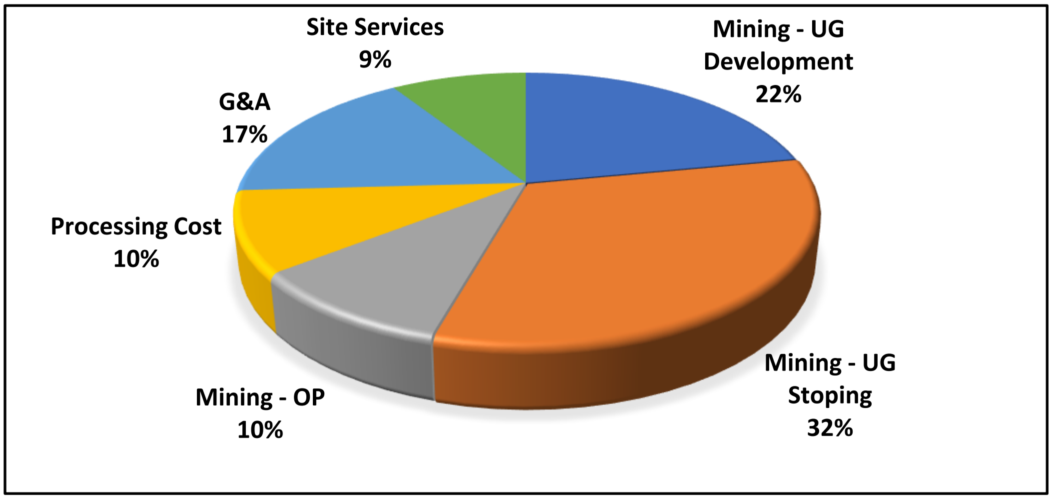
The cost estimates in this section are based upon the consumable prices and labour salaries/wages in Q3 2025 or based on the information from the Tetra Tech's database. Where applicable, costs in this estimate have been converted from US Dollars to Canadian Dollars using the currency exchange rate of CAD1.00:USD0.72. The expected accuracy range of the operating cost estimate is within ±35%.

Table 21-3: LOM Average Operating Cost Summary

Function	Operating Cost (C$/t processed)
Mining Cost - UG-Development	40.71
Mining Cost - UG-Stoping	60.13
Mining Cost - OP	18.59
Processing Cost	17.96
G&A	31.23
Site Services	16.76
Total Operating Cost	**185.38**

Note: 1. LOM average operating at 900 t/d, which is slightly different from the unit cost at the nominal process rate of 1,000 t/d. 2. G&A includes worker freight costs and catering costs.

Figure 21-1: LOM Average Operating Cost Distribution by Operation Unit



21.3.2 Mining Operating Cost

The estimated mining operating cost is based on contracted mining services. Table 21-4 outlines the LOM average cost per tonne mined by mining activity.

Table 21-4: Life of Mine Average Mining Operating Cost

Area	LOM Average Unit Cost, C$/t Mined*
UG Development	78.00
UG Stoping	93.00
OP Mining	7.00
Overall (UG + OP) Average	**36.25**

*Costs are per tonne mined, which are different than the costs per tonne processed in Table 21-3

21.3.3 Processing Operating Cost

The average annual operating cost for the nominal processing rate of 1,000 t/d (900 t/d LOM average, 12-hour per day operation) is estimated to be $5.8 million per year, or $16.00/t processed. The processing operation is day shift only or 12 hours per operating day. This is equivalent to a LOM average operating cost of $17.96/t processed. This estimate includes:

- Staffing and salary/wage level estimates, based on the 2025 Q3 mining industry labour rates in BC.
- Power consumption estimates are based on equipment load list power draw estimates.

- Unit power cost estimate of $0.35/kWh based on site power generation.
- Operating and maintenance supply cost estimate, based on approximately 7% to 8% of major equipment capital costs, benchmarked against comparable operations in BC and worldwide.
- Includes the assay laboratory operation on Project site.
- No taxes have been accounted in the estimates, unless specified.

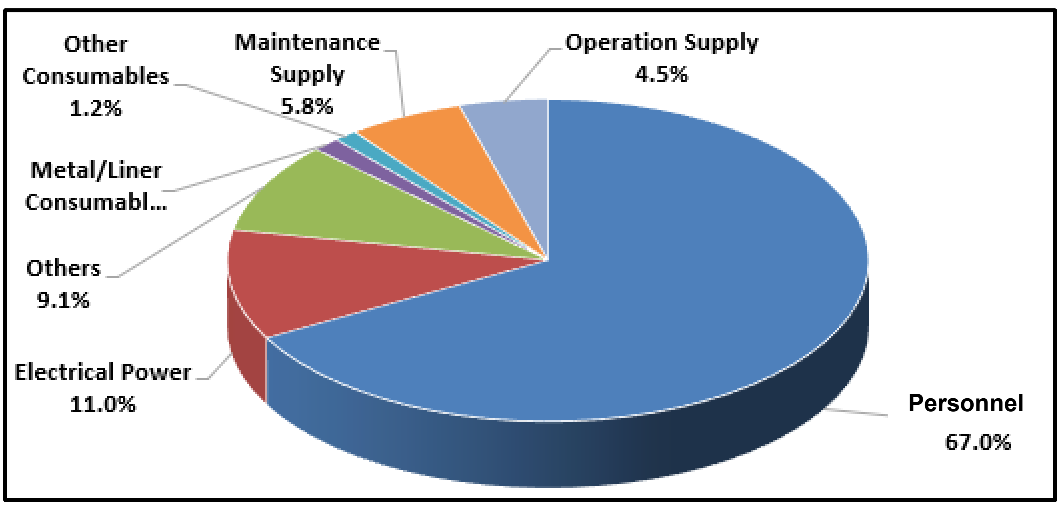
Table 21-5 summarizes the estimated processing costs. Figure 21-2 shows the average operating cost distribution by area.

Table 21-5: Summary of Processing Costs

Description	Personnel	Annual Cost (C$)	Unit Cost (C$/t plant feed)	Unit Cost (C$/t product)
Personnel	28	3,912,000	10.72	18.96
Consumables		152,000	0.42	0.74
Maintenance Supplies		341,000	0.93	1.64
Operating Supplies		261,000	0.71	1.26
Power Supply		644,000	1.76	3.11
Others (Contingency)		531,000	1.45	2.56
Total		**5,840,000**	**16.00**	**28.27**

Note: Rounded to the nearest cent.

Figure 21-2: Average Processing Operating Cost Distribution by Area



Power, consumable and supply costs constitute approximately 26% of total processing costs. The cost estimates are detailed in the following sections.

Processing Operating Cost

Labour Cost

Total processing labour costs at the nominal processing rate of 1,000 t/d are estimated to be $3.9 million per year, or $10.72/t of crushing feed. Salary estimates are based on 2025 labour rates for the 2025 mining industry labour rates in BC, Canada. The comminution and sorting plant will be staffed with 28 personnel including 5 in general supervisory and technical services, 14 in operational roles, and 9 in maintenance roles. The operation staff schedule is based on 2-weeks in/2-week out rotation, dayshift only. Approximately half of the staff will be on site at any given time.

Loaded salary includes base salary and burden, which is estimated to be 40 % of the base payment. The burden includes holiday and vacation payments, overtime payments, pension plan, Medical Services Plan, Canada Pension Plan, Employment Insurance, Workers' Compensation Board insurance and tool allowance costs.

Power Cost

The average annual processing power consumption, based on the nominal processing rate of 1,000 t/d, is estimated to be 1.82 MWh/a. At an average power unit cost of $0.35/kWh, the annual power cost is estimated to be $644,000, or $1.76/t of crushing feed.

Maintenance and Operating Supplies Cost

Maintenance and operating supplies are estimated to cost $602,000 per year, or $1.64/t of crushing feed. Maintenance supplies are estimated to cost $341,000 per year or $0.93/t of crushing feed. Operating supplies are estimated to be $261,000 per year, or $0.71/t of crushing feed.

Major Consumable Costs

Major consumable costs are estimated to be $152,000 per year at the nominal processing rate of 1,000 t/d, or $0.42/t of plant feed.

The estimates of major consumable costs are based on the following:

- Consumption rates for crusher liners are based on the data provided by the potential crusher suppliers, or in-house database.
- Required screen panels, dust collector media and others are based on Tetra Tech’s database.

No taxes have been accounted in the estimates, unless specified.

21.3.4 General and Administrative

For the LOM average, G&A expenses are estimated at approximately $9.6 million per year, or $31.23/t processed. The costs include:

- Personnel – supervisors and staffing in accounting, purchasing, environmental departments, and other G&A departments.
- G&A expenses including insurance, administrative supplies, medical services, legal services, human resources related expenses, travelling, electricity power demand charge, accommodation/camp costs, air and bus crew transportation, and external assay/testing.
- Labour cost and general expenses are estimated to be approximately $10.64/t processed and $20.59/t processed, respectively. The major costs are accommodation and crew air and bus transportation, estimated at approximately $4.3 million per year.

21.3.5 Surface Services

The LOM average site service cost is estimated at $15.07/t processed or approximately $4.7 million per year. The estimate includes:

- Personnel – general surface services labour
- Surface mobile equipment and light vehicle operations
- General snow removal/road maintenance
- Potable water and waste management
- General maintenance including yards, roads, fences, and building maintenance
- Off-site operation expense
- Building heating
- Avalanche control

21.3.6 Water Treatment

The LOM total cost for water treatment is estimated at $3.7 million, based on very limited information available. This cost will be reviewed and revised during the next phase of the study and after the completion of baseline data collection, hydrogeological drilling, and water quality modelling.

22.0 ECONOMIC ANALYSIS

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

22.1 Introduction

The economic analysis of The Project has been derived from the inputs described in previous chapters. The economic analysis has been performed on a 100% basis using Q3 2025 Canadian dollars, and its unlevered post-tax FCF has been discounted using mid year discounting at a rate of 5% per annum. NSR, capital, operating, sustaining and closure costs, Net Profits Interest (NPI) payments, BC Mineral tax, and Federal and Provincial income taxes are included in the financial analysis.

The following investment returns and select financial metrics presented in Table 22-1 were calculated based on three gold price scenarios and assumptions listed in Section 22-2.

Table 22-1: Summary of Economic Analysis Results

Base Case – Gold Price at US$2,600	**NPV$^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$326.1M	82.5%	1.0 year	2.5
After-Tax	$215.8M	60.3%	1.2 year	1.7
Alternate Case 1 – Gold Price at US$3,400	**NPV$^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$681.2M	148.9%	0.6 year	5.3
After-Tax	$442.0M	107.9%	0.8 year	3.4
Alternate Case 2 – Gold Price at US$4,200	**NPV$^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$1,036M	212.1%	0.5 year	8.1
After-Tax	$668.3M	153.2%	0.6 year	5.2

22.2 Inputs/Assumptions to the Preliminary Financial Analysis

The inputs to the preliminary economical assessment include:

- Mine production schedule, including dilution and waste rock handling

- Process plant feed sorting recovery rates to develop the annual recovered metals based on metallurgical test work results
- Initial capital expenditures, including required construction and development before commercial production, including mining development and overall project construction expenditures as specified in Section 21.0
- Sustaining capital costs on a year-by-year basis over the LOM, including the $15 million mine closure and reclamation costs distributed over the LOM
- Opex, including the costs for mining, processing, general administration, overall site services, and site water management
- Off-site expenditures, such as applicable treatment, refining, transportation, and insurance costs (the offsite charges are detailed in Section 22.3)
- Metal prices as noted and the foreign exchange rate of CAD1.00:USD0.72
- Royalties detailed in Section 22.4

Annual at-mine revenue contribution of the metal was determined by deducting the off-site charges from gross revenue. The financial model also includes working capital that will be recovered during the LOM.

Metal price and foreign exchange rate inputs to the financial model are flat across all years.

The financial analysis does not include sunk costs and any costs associated with financing. In addition, all the cash flows in the financial model were based on Q3 2025 Canadian dollars without any adjustment for inflation.

The years used for the preliminary financial analysis are for illustrative purposes only and do not necessarily represent a commitment to the start dates or actual production.

22.3 Smelter Terms and Transport Costs

Smelter terms are defined for the existing resource through an agreement with Ocean Partners. The transportation costs are estimated bases on quotations and in house data.

22.3.1 Smelting Terms

The Project Smelting Terms are shown in Table 22-2.

Table 22-2: Smelting Terms

Item	Unit	Value
Payability - Gold	%	86.5% to 89.0% LOM Average 88.3%
Treatment Charge:		
Base	US$/t dry product	70.00

Item	Unit	Value
Additional	US$/t dry product	65.00
Refining Charge	US$/t payable Au oz	6.00

22.3.2 Transportation Cost and Insurance Estimates

The estimated transportation costs for the product and the costs related to insurance are as follows in Table 22-3.

Table 22-3: Transportation Cost and Insurance Assumptions

Item	Unit	Value
Product Moisture Content	%	3.0
Product Freight from Project Site to Seaport in Stewart, BC	C$/t wet product	26.03
Port Handling	C$/t wet product	33.50
Ocean Shipping	US$/t wet product	50.00
Insurance	of invoiced product value	0.125%
Product Transport Loss	of product value	0.10%

22.4 Royalty

Franco-Nevada Corporation currently holds a 2.0% gross production royalty on all claims on the project. This is the only royalty on the Scottie Gold Mine Project, all historic royalties on the Scottie Project have been eliminated (MMTS, 2025).

The claims/crown grants are subject to a 2% Gross Production Royalty held by Franco-Nevada. No other royalty or encumbrance exists on the claims (MMTS, 2025).

22.5 Assumptions on Taxes

The following general tax regime was recognized as applicable at the time of report writing:

22.5.1 Canadian Federal and BC Provincial Income Tax Regime

Federal and BC provincial income taxes are calculated using the currently enacted corporate rates of 15% for federal and 12% for BC.

For both federal and provincial income tax purposes, capital expenditures are accumulated in pools that can be deducted against mine income at different rates, depending on the type of capital expenditure.

Mineral exploration costs are accumulated in the Canadian Exploration Expense (CEE) pool. The CEE pool is generally amortized at 100%, to the extent of taxable income from the mine.

Resource property acquisition costs and most pre-production mine development costs are accumulated in the Canadian Development Expense (CDE) pool. The CDE pool is generally amortized against income at 30% on a declining balance basis.

Fixed assets used in mining operations are accumulated in an Undepreciated Capital Cost pool, Class 41. The Class 41 pool is amortized at 25% on a declining balance basis.

22.6 BC Mineral Tax Regime

The BC Mineral Tax regime is a two-tier tax regime, with a 2% tax and a 13% tax.

The 2% tax is assessed on “net current proceeds”, which is defined as gross revenue from the mine less mine Opex. Hedging income and losses, royalties and financing costs are excluded from Opex. The 2% tax is accumulated in a Cumulative Tax Credit Account (CTCA) and is fully creditable against the 13% tax.

All capital expenditures, both mine development costs and fixed asset purchases, are accumulated in the Cumulative Expenditures Account, which is amortized at 100% against the 13% tax.

The 13% tax is assessed on “net revenue”, which is defined as gross revenue from the mine, less mine Opex, less any accumulated cumulative expenditures account balance. As such, the 13% tax is not assessed until all pre-production capital expenditures have been amortized.

“new mine allowance” is provided for capital costs of new mines to encourage mine investment in BC. This allowance provides that 133% of capital expenditures incurred prior to commencement of production are included in the cumulative expenditures account. Under current legislation, the provision for the new mine allowance applies to mines that begin producing minerals before December 31, 2030. The post-tax model is calculated on the assumption that the mine allowance will apply.

Notional interest of 125% of the prevailing federal bank rate is calculated annually on any unused cumulative expenditures account and CTCA balances and is added to these pools.

BC Mineral Tax is deductible for federal and provincial income tax purposes.

22.7 Financial Model Summary

The production schedule has been incorporated into the 100% equity, pre-tax financial model to develop annual production based on tonnage sorted, head grades, and recoveries.

Metal revenues are derived from gold sales. Operating cost for mining, processing, site services, G&A, as well as off-site charges (smelting, refining, transportation) were deducted from the revenues to derive annual operating cash flow.

Initial and sustaining capital costs as well as closure and reclamation costs have been incorporated on an annual basis over the mine life and deducted from the operating cash flow to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of concentrate, including all pre-production mining costs. Sustaining capital includes expenditures for mining and processing additions, replacement of equipment, and infrastructure expansions.

Initial and sustaining capital costs applied in the economic analysis are $128.6 million and $76.7 million, respectively. The pre-production construction period is estimated to be one year. NPV and IRR are estimated at the start of this one-year period.

Working capital is included in the financial analysis and varies from year to year. The working capital is recovered at the end of the mine life.

Table 22-4 shows the cash flow and the key economic parameters.

Table 22-4: Economic Analysis Summary (2025 PEA Base Case)

Description	Value	Units
Gold Price	2,600	US$/oz
Canadian Dollar to US Dollar Exchange Rate	0.72	C$:US$
Average Processing Throughput	900	t/d
Mine Life	7	year
Milled Tonnage	2.19	Mt
Average LOM Gold Head Grade	6.86	g/t
Contained Gold	483,000	oz
Gold Recovery	94.7	%
Payable Gold, LOM	457,600	oz
Average Annual Gold Production (LOM)	65,400	oz
Average Annual Gold Production (Year 1 to 4)	77,300	oz
Total Operating Cost	185.38	$/t processed
UG Mining Cost	118.10	$/t processed
OP Mining Cost	6.95	$/t mined
Processing Cost	17.96	$/t processed
G&A Cost	31.23	$/t processed
Surface Services Cost, including Water Management	16.76	$/t processed
Initial Capital Cost	128.6	$ million
LOM Sustaining Capital Cost	76.7	$ million
LOM AISC	1,452	US$/oz Au
Before-Tax IRR	82.5	%
Before-Tax NPV	326.1	$ million, 5% discount rate
Before-Tax Undiscounted LOM net free cash flow	419.1	$ million
Before-Tax Payback period	1.0	year

Description	Value	Units
After-Tax IRR	60.3	%
After-Tax NPV	215.8	$ million, 5% discount rate
After-Tax Undiscounted LOM net free cash flow	283.5	$ million
After-Tax Payback period	1.2	year

The project's post-tax FCF summary is shown in Figure 22-1.

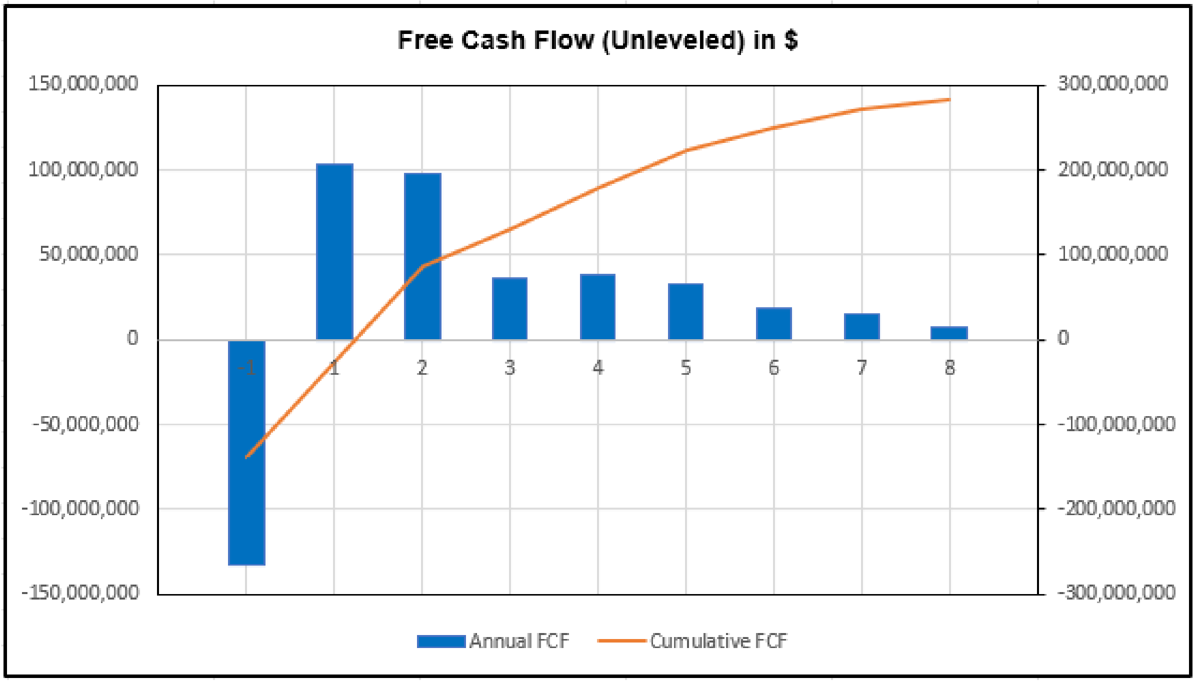


Figure 22-1: Post-Tax Annual and Cumulative FCFs

Federal, Provincial, and BC Mineral Tax payables based on the PEA financial model are calculated and shown below in Table 22-5. The BC Mineral Tax (Provincial Resource Tax) is deductible from Federal and Provincial taxes payable.

Table 22-5: Project Cash Flow Summary

Gold Price (US$/oz)	2,600	3,400	4,200
Net Cash Flow (Before-tax, $ millions)	419.1M	857.8M	1,296M
Net Cash Flow (After-tax, $ millions)	283.5M	561.9M	840.5M
NPV @5% (Before-tax, $ millions)	326.1M	681.2M	1,036M
NPV @5% (After-tax, $ millions)	215.8M	442.0M	668.3M

22.8 Sensitivity Analysis

Sensitivity analysis for after-tax financial parameters for both NPV at a discount rate of 5% and IRR were conducted and are presented in Figure 22-2 to Figure 22-5. The financial parameters include:

- Gold price
- Exchange rate
- Development capital expenditure
- Operating costs

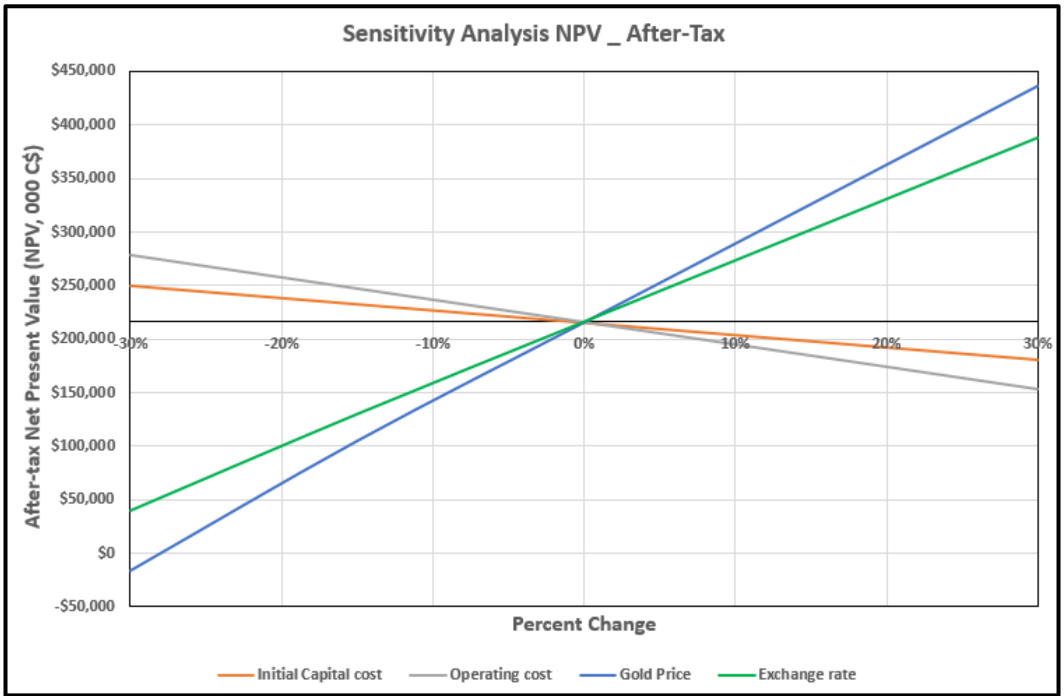


Figure 22-2: After-tax NPV Sensitivity (+/-30%)

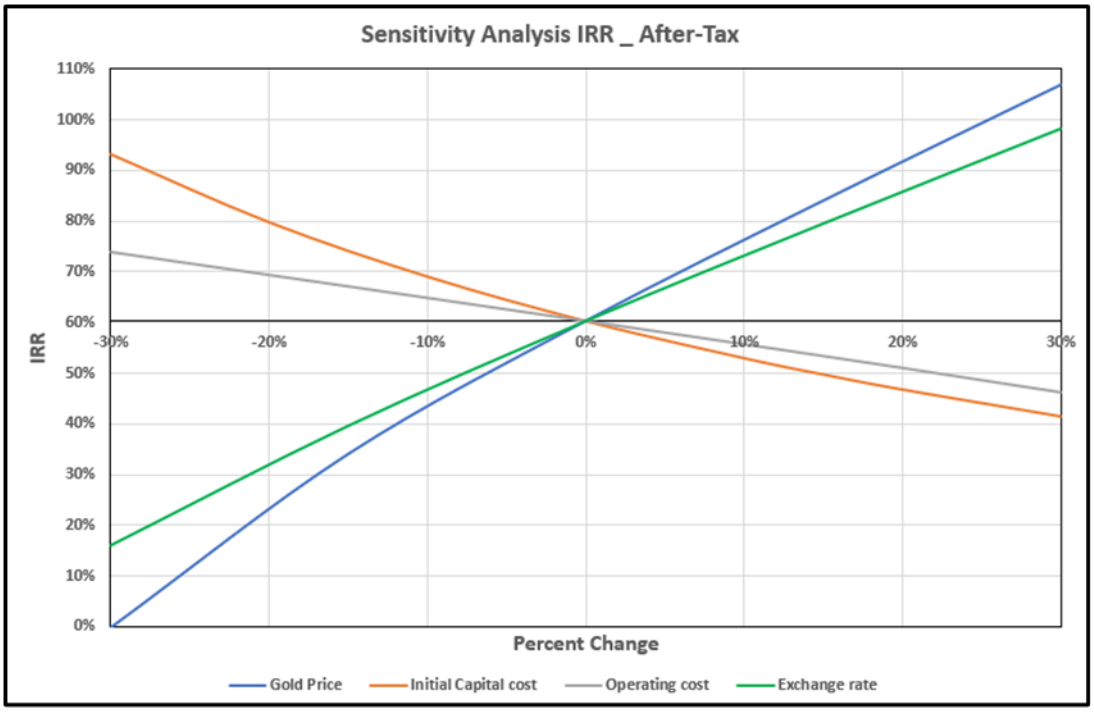


Figure 22-3: After-tax IRR Sensitivity (+/-30%)

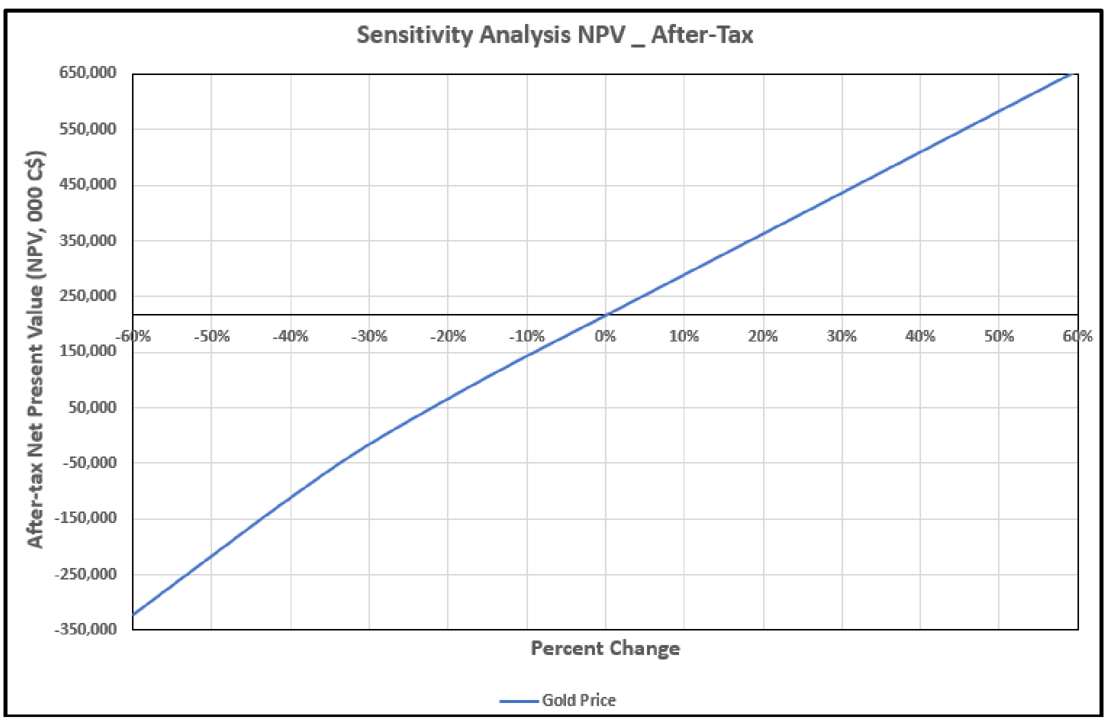


Figure 22-4: After-tax NPV Sensitivity (Gold Price Only, +/-60%)

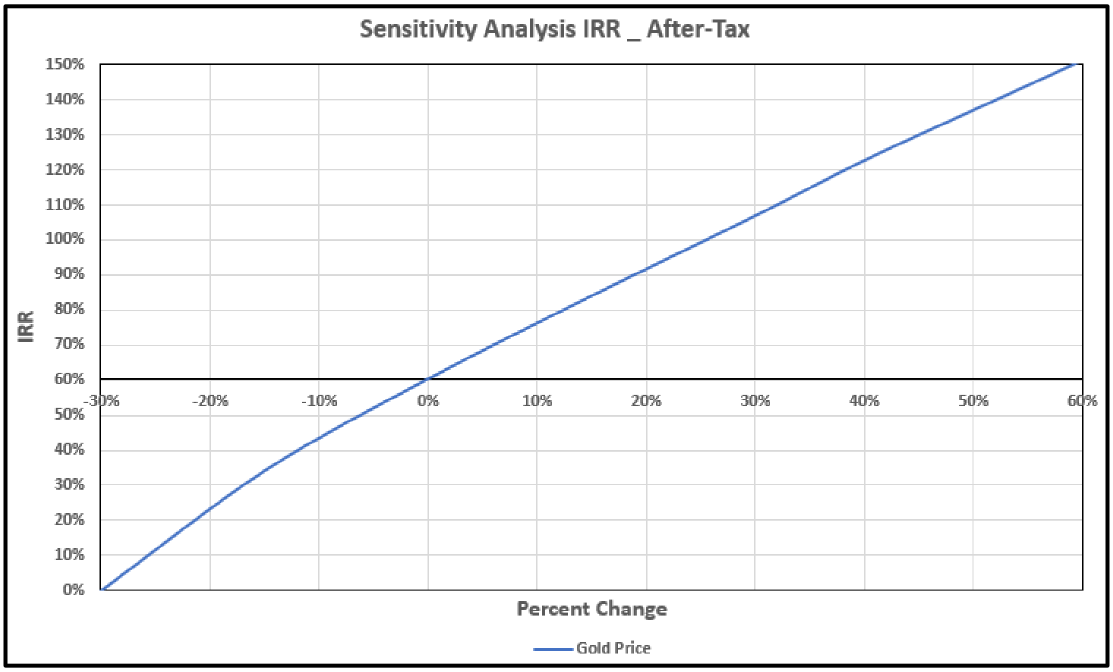


Figure 22-5: After-tax IRR Sensitivity (Gold Price Only, +/-60%)

23.0 ADJACENT PROPERTIES

sc ot Resource’s Premier Gold Project (PGP) is located 25 km north of the town of Stewart, British Columbia, about 20km south of the Scottie Gold Mine Project along the Granduc Mine Road. Three of the deposits are based at PGP and the fourth deposit is located at the Red Mountain.

Project situated approximately 23 km to the southeast of the PGP mill. The current Feasibility Study (Ascot, 2020) is based on four underground mining operations feeding a centralized 2500 tpd processing facility, located at PGP. The four mining operations known as Silver Coin, Big Missouri, Premier and Red Mountain will be sequenced over an 8-year period to initially produce 1.1 Moz. of gold and 3.0 Moz. of silver. PGP benefits from existing road access, historical mining, milling, the nearby Long Lake Hydro power plant, tailings and mine waste stockpile infrastructure resulting in a low initial capital refurbishment cost.

24.0 OTHER RELEVANT DATA AND INFORMATION

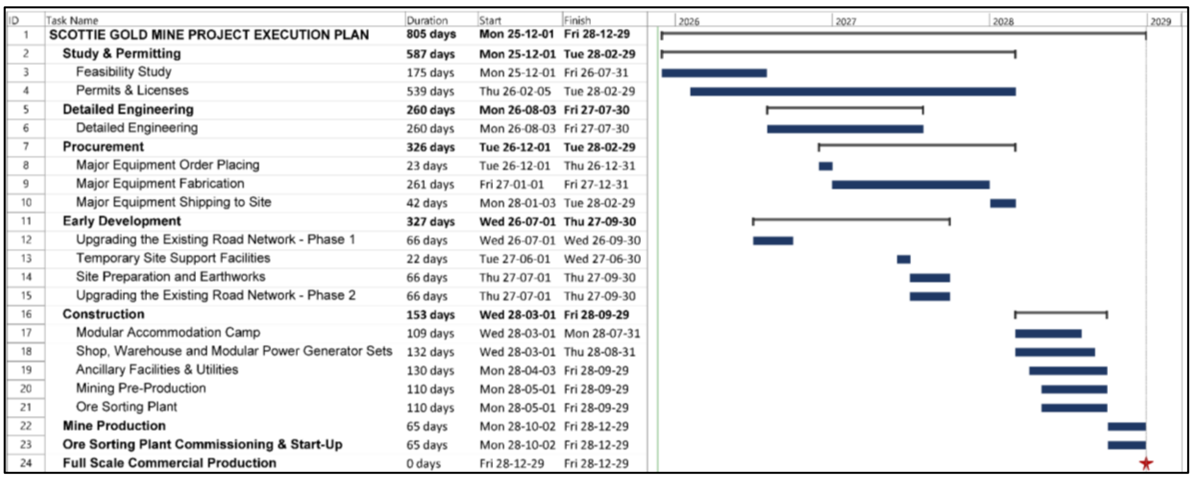
24.1 Project Execution Schedule

A preliminary project execution plan has been prepared for the PEA, which describes a logical sequence by which the Project may advance from its current status to the start of mine production.

Environmental studies and permitting (discussed in Section 20.0) will accompany and enable the engineering and construction work described in this section.

Scottie's current activities at the Project site continue to demonstrate full compliance with all applicable regulations, maintenance of a safe and healthy working environment, care and concern for the natural environment, and responsiveness to the needs and ambitions the Indigenous and northern communities. Scottie will maintain and enhance these standards as the Project develops towards mine production.

The planned development of the Project is summarized in Figure 24-1.



ID	Task Name	Duration	Start	Finish
1	**SCOTTIE GOLD MINE PROJECT EXECUTION PLAN**	**805 days**	**Mon 25-12-01**	**Fri 28-12-29**
2	**Study & Permitting**	**587 days**	**Mon 25-12-01**	**Tue 28-02-29**
3	Feasibility Study	175 days	Mon 25-12-01	Fri 26-07-31
4	Permits & Licenses	539 days	Thu 26-02-05	Tue 28-02-29
5	**Detailed Engineering**	**260 days**	**Mon 26-08-03**	**Fri 27-07-30**
6	Detailed Engineering	260 days	Mon 26-08-03	Fri 27-07-30
7	**Procurement**	**326 days**	**Tue 26-12-01**	**Tue 28-02-29**
8	Major Equipment Order Placing	23 days	Tue 26-12-01	Thu 26-12-31
9	Major Equipment Fabrication	261 days	Fri 27-01-01	Fri 27-12-31
10	Major Equipment Shipping to Site	42 days	Mon 28-01-03	Tue 28-02-29
11	**Early Development**	**327 days**	**Wed 26-07-01**	**Thu 27-09-30**
12	Upgrading the Existing Road Network - Phase 1	66 days	Wed 26-07-01	Wed 26-09-30
13	Temporary Site Support Facilities	22 days	Tue 27-06-01	Wed 27-06-30
14	Site Preparation and Earthworks	66 days	Thu 27-07-01	Thu 27-09-30
15	Upgrading the Existing Road Network - Phase 2	66 days	Thu 27-07-01	Thu 27-09-30
16	**Construction**	**153 days**	**Wed 28-03-01**	**Fri 28-09-29**
17	Modular Accommodation Camp	109 days	Wed 28-03-01	Mon 28-07-31
18	Shop, Warehouse and Modular Power Generator Sets	132 days	Wed 28-03-01	Thu 28-08-31
19	Ancillary Facilities & Utilities	130 days	Mon 28-04-03	Fri 28-09-29
20	Mining Pre-Production	110 days	Mon 28-05-01	Fri 28-09-29
21	Ore Sorting Plant	110 days	Mon 28-05-01	Fri 28-09-29
22	**Mine Production**	65 days	Mon 28-10-02	Fri 28-12-29
23	**Ore Sorting Plant Commissioning & Start-Up**	65 days	Mon 28-10-02	Fri 28-12-29
24	**Full Scale Commercial Production**	0 days	Fri 28-12-29	Fri 28-12-29

Figure 24-1: Scottie Gold Mine Project High Level Execution Plan

The preliminary project execution schedule (the Schedule) was developed to provide a high-level overview of all activities required to complete the project.

Upon receipt of construction and operating permits, the project is expected to take approximately two years to complete, from the time board approval is received, through construction, commissioning and commencement of production.

The critical path of the project schedule is composed of activities related to:

1. Feasibility Study
2. Baseline Studies and Environmental Application
3. Permitting and Licensing

TETRA TECH

4. Detailed Engineering
5. Construction
6. Commissioning

Additional activities such as trade-off studies, additional drilling programs, metallurgical testing, as well as major equipment fabrication can proceed in parallel to the critical path activities.

24.2 Alternative Mineral Processing and Gold Extraction Scenario

Of the studied project design components, the PEA demonstrates that the most profound improvement to the project economics is toll milling the product at the nearby Premier mill. No toll milling arrangement is currently in place with the Premier mill, it is considered in the PEA as a recommendation for further study work. The Premier mill is located halfway along the trucking route to the Stewart Terminal. The model assumes appropriate operating costs derived from scot 's 2020 Feasibility Study, with an additional toll milling premium applied, similar to comparable toll milling projects. At US$2600/oz gold the AISC for the toll milling model is calculated to be US$935 (versus US$1452 in the base-case DSO model). The results of the economic analysis performed for the toll mill scenario are presented in Table 24-1. The PEA base case (DSO) economic analysis results are presented next to the toll milling option economic analyses results for comparison.

Table 24-1: Gold Price Sensitivity and Comparison Table between DSO Base Case and Toll Milling Option for the Scottie Gold Mine Project

Gold Price	Description	Toll Milling Option*	DSO Base Case
US$2,600/oz	After-Tax NPV5%	$380.1M	$215.8M
	After-Tax IRR	89.9%	60.3%
	After-Tax Payback	0.9 years	1.2 years
	After-Tax NPV/CAPEX	3.0	1.7
US$3,400/oz	After-Tax NPV5%	$606.0M	$442.0M
	After-Tax IRR	135.2%	107.9%
	After-Tax Payback	0.7 years	0.8 years
	After-Tax NPV/CAPEX	4.7	3.4
US$4,200/oz	After-Tax NPV5%	$831.7M	$668.3M
	After-Tax IRR	177.5%	153.2%
	After-Tax Payback	0.5 years	0.6 years
	After-Tax NPV/CAPEX	6.5	5.2

Note: Scottie Gold Mine Preliminary Economic Assessment Base Case assumes a gold price of US$2600/troy ounce ("oz") and a US$/CAD$ exchange rate of 0.72:1.00. NPV/CAPEX is the ratio between NPV value versus Initial Capex *At this time there is no toll milling arrangement in place with the nearby Premier mill.

Additional opportunities of refinement on this concept include: (1) optimizing mine plan and resource for increasing more gold ounces for the treatment (i.e., include more lower grade ounces), (2) elimination of the ore sorting plant, and (3) further metallurgical test work on the mineralization to maximize recovery in a toll milling scenario. In the PEA, the cyanide leaching recoveries is set at 89.1%. Historic production and recent test work suggests potential for improved gold recoveries to approximately 91% to 95%. The results from the preliminary metallurgical test work also suggested that a finer grinding would increase leaching recoveries of up to 97% for both gold and gold gravity concentrates. Scottie intends to follow up these promising results with further test work to be completed and incorporated into the Feasibility Study (FS).

25.0 INTERPRETATION AND CONCLUSIONS

The Scottie Gold Mine Project is considered to be technically and economically viable based on the results of the work presented in this Technical Report. It is recommended to advance the Project to the next stage, Feasibility Study.

25.1 Summary

The environmental setting is a key factor in the current development strategy, which aims for a minimal environmental impact footprint. The Property is situated approximately 35 km north of Stewart, BC, in a remote but road-accessible area along the Granduc Road. The property lies within a geologically favorable position, hosting Late Triassic Stuhini Group volcanic and sedimentary units and Early Jurassic Hazelton Group volcanic and volcano-sedimentary units. The gold mineralization is primarily hosted in andesitic volcanic rocks adjacent to a contact with an Early Jurassic intrusion known as the Texas Creek Plutonic Suite.

The project benefits from existing infrastructure, including proximity to the Brucejack Transmission Line, a deep-water shipping port, and existing road networks, which helps minimize new surface disturbance. Scottie Resources Corp., the current owner, has emphasized minimizing the project's environmental impact through its development plans. The company initiated comprehensive environmental baseline studies in 2025 as a critical step in the permitting process. These studies typically cover water quality, air quality, ecosystem health, and other environmental factors.

DSO Model: A key aspect of the project's PEA is a DSO model, which plans to ship raw, upgraded mineralization to off-site processing facilities. This design is intended to eliminate the need for an on-site gold processing plant and large tailings facilities associated with conventional milling operations.

By utilizing existing infrastructure and the DSO model, the project aims for a low-impact operation with minimal new construction and a smaller overall environmental footprint. As a past-producing mine, there may be some legacy issues from historical operations, but recent efforts are focused on modern and responsible development practices.

Overall, the environmental setting is that of a mineral-rich, mountainous region, with current development efforts prioritizing a project design that is physically small and minimizes long-term environmental risk.

25.2 Mineral Resource Estimate, Sampling, Preparation, Analysis, and Data Verification

The procedures documented for sampling, analysis and security are deemed adequate. Analysis of the QAQC samples indicates the laboratory results are of sufficient quality for resource estimation. Inconsistencies detected during validation of the assay database are minimal.

In the opinion of the QP the block model resource estimate and resource classification reported herein are a reasonable representation of the global gold mineral resources found in the Scottie and Blueberry deposits. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

25.2.1 Sampling, Preparation, Analysis and Data Risks and Opportunities

An opportunity exists for Scottie Gold to add QAQC data for silver and to collect and complete the missing certificate numbers in the database. This information would more completely support the assay database.

25.2.2 Resource Estimate Risks and Opportunities

Risk in the geologic interpretations relating to the continuity of mineralization exist and can be mitigated by additional geologic modelling for use in controlling the block model interpolations. A description of additional potential risk factors concerning the resource estimate is given in Table 14-13 along with either the justification for the approach taken or mitigating factors in place to reduce any risk.

Opportunities to increase confidence in the resource through infill drilling and to expand the resource from step-out and exploration drilling are discussed in the recommendations section below.

25.3 Mining Methods

The Project is planned as a combined open pit (OP) and underground (UG) mining operation. Open pit development is expected to utilize a conventional truck-and-shovel method, whereas underground development will be based on longitudinal longhole stoping. A nominal average production rate of approximately 1,000 tonnes per day (tpd) has been assumed for the combined operation.

For the base case scenario, run-of-mine (ROM) plant feed will be stockpiled and subsequently processed using an ore sorter. The ore-sorter concentrate will be shipped directly to overseas markets. This mining and processing strategy has been applied to both the open pit and underground components and forms the basis of the PEA.

Factors which may affect the mine plan include changes to the geotechnical parameters, gold price, exchange rate, operating costs, marketing assumptions, and metallurgical recoveries.

The annual mine schedule for the combined open pit and underground mining is shown in Table 25-1.

Table 25-1: Mine Schedule

Description	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Total ore-sorter feed tonnage	(kt)	2,186	199	352	326	341	380	318	270
Grade – Au	(g/t)	6.9	7.4	8.2	7.8	7.0	5.8	5.4	6.6
Metal content – Au	(koz)	482	48	93	82	77	71	55	57

25.4 Metallurgical Testing and Recovery Methods

The recent metallurgical test works since 2024 used for the PEA process design are based on mill feed pre-concentration using ore-sorting and dense media separation (DMS), including the latest ore-sorting metallurgical test work conducted during 2024/2025. It appears that the test samples responded reasonably well to the sorting preconcentration treatment. The test results also show that the samples tested should be amenable to the gold extraction by cyanidation.

Based on the results of the test work, a cost-effective processing flowsheet was developed to recover and upgrade the plant feed into a saleable concentrate using ore-sorting technique. The processing plant includes the following unit operations:

- A primary crusher operates in closed-circuit with a triple-deck dry screen.
- Coarse product from the dry screen is washed using a double-deck wet screen to prepare the feed for two ore sorters. The undersize from the dry screen is stockpiled and later blended with the sorter concentrates.
- The two fractions (coarse & fine) from the wet screen are sorted separately using two X-ray fluorescence (XRF) sorters to produce final concentrate, which is shipped directly overseas for sale.
- Wash water from the wet screen is recycled using a belt filter.
- All the associated utilities required for plant operation are included.

The processing plant will operate at a nominal throughput of 1,000 t/d, producing gold concentrate. The LOM average plant feed grade is expected to be 6.86 g/t Au, and the anticipated average recovery will be 94.7%, including the fines blending with sorter concentrate. The LOM average annual concentrate production will be approximately 177,000 t/y at 11.5 g/t Au.

25.5 Infrastructure

The Project will require the development of a number of infrastructure items. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and for efficient and convenient operation of the mine equipment fleet. Buildings will be equipped with high pitched roofs for efficient snow clearing.

The Project surface infrastructure will include the following:

- Processing: An ore-sorting plant with stockpiles and product loadout.
- Accommodation: A modular 160-person accommodation camp with arctic corridors.
- Equipment maintenance and storage facilities: A surface mobile equipment shop and warehouse with mine dry and offices, a cold storage warehouse, a surface maintenance laydown and storage area and detonator and explosive storage magazines.
- Utilities: A power plant with a 13.8kV power distribution system, water supply and distribution, fuel storage and distribution, site communication system, waste management facilities, sewage treatment facility, surface water management structures.

- Storage areas for sorted products and waste rocks, and access roads for connecting site infrastructure and facilities.

25.6 Capital and Operating Costs

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $128.6 million. A summary breakdown of the initial capital cost is provided in Table 25-2. This total includes all direct costs, indirect costs, wner's costs, and contingencies. ll costs are shown in Canadian Dollars unless otherwise specified.

Table 25-2: Initial Capital Cost Summary

Description	Initial Capital Cost (C$ million)
Direct Costs	
Mining Infrastructure	6.8
Site Preparation	5.0
Sorting Plant	26.2
On-Site Facilities: Camp, Power Plant, Laboratory, and Other Facilities	28.5
Surface Mobile Equipment	7.8
Utilities, such as Fresh/Potable Water, Power Distribution and Waste Management	7.3
Water Management and Avalanche Control	3.0
Subtotal – Direct Costs	**84.6**
Indirect Costs	
Project Indirect Costs	23.7
wner' s Costs	3.4
Contingencies	16.9
Total	**128.6**

On average, the LOM on-site operating costs for the Project were estimated to be $185.38/t of the plant feed. The operating costs are defined as the direct operating costs including mining, processing, surface services, and G&A costs (Table 25-3).

Table 25-3: LOM Average Operating Cost Summary

Function	Operating Cost (C$/t product)
Mining Cost - UG-Development	40.71
Mining Cost - UG-Stoping	60.13
Mining Cost - OP	18.59
Processing Cost	17.96
G&A	31.23
Site Services	16.76
Total Operating Cost	**185.38**

Note: 1. LOM average operating at 900 t/d, which is slightly different from the unit cost at the nominal process rate of 1,000 t/d.. 2. G&A includes worker transport freight and catering costs

25.7 Economic Analysis

For the seven-year mine life and 2.19 Mt processing plant feed tonnage, 1.24 Mt sorted concentrate tonnage, and the foreign exchange rate of CAD1.00:USD0.72, the following investment returns and select financial metrics presented in Table 25-4 were calculated based on three gold price scenarios.

Table 25-4: Summary of Economic Analysis Results (PEA Base Case and Two Alternate Cases)

Base Case – Gold Price at US$2,600	**$NPV^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$326.1M	82.5%	1.0 year	2.5
After-Tax	$215.8M	60.3%	1.2 year	1.7
Alternate Case 1 – Gold Price at US$3,400	**$NPV^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$681.2M	148.9%	0.6 year	5.3
After-Tax	$442.0M	107.9%	0.8 year	3.4
Alternate Case 2 – Gold Price at US$4,200	**$NPV^{5\%}$**	**IRR**	**Payback**	**NPV/Initial Capex**
Before-Tax	$1,036M	212.1%	0.5 year	8.1
After-Tax	$668.3M	153.2%	0.6 year	5.2

26.0 RECOMMENDATIONS

26.1 2026 Feasibility Study

The Scottie Gold Mine Project is considered to be technically and economically viable based on the results of the work presented in this Technical Report. It is recommended to advance the Project to the next stage, Feasibility Study.

26.2 Mineral Resource Estimate and Geology

The resource estimated for the PEA is based on the February 2025 Mineral Resource Estimate, which includes the Blueberry Zone, Scottie Gold Mine, and Bend vein. With success on further drilling, there are several ways that expanded resources could improve the economics of the project, including higher throughput, extended mine life, and bringing in additional isolated stopes left off the PEA mine plan due to development costs.

For sample Preparation, Analyses and Security, MMTS recommends that additional check assays be collected for QAQC analyses to ensure and further improve data quality.

For Data Verification, future drilling should include third party check-assays, and the data should be appropriately maintained.

MMTS also recommends continuing drilling the deposits to upgrade the Classification to Indicated by Infill drilling and to expand the resource. Each are of the project has expansion potential as they are all currently open along strike and at depth.

26.3 Geotechnical

The FS geotechnical drilling program has been largely completed in 2025; there are several more drill holes to be completed in 2026. Hydrogeological information will also be collected by conducting testing in these geotechnical drill holes.

26.4 Mining

Tetra Tech makes the following recommendations for future mining work:

- The Project should proceed to the next level of study. A detailed mining production schedule and design should be developed with detailed mining activities to understand the potential constraints and cost reduction opportunities.
- An underground survey and interpretation of the Scottie historical workings.
- As the pit optimization and scheduling results are highly dependent on the geotechnical parameters, more detailed geotechnical studies and/or fieldwork should be conducted to better define the appropriate pit slope angles and design parameters for the pit, stockpile, and waste dump

- To estimate pit dewatering requirements, a hydrogeological study should be completed.
- A detailed characterization of mine waste material should be completed to enhance the waste management.
- Reduced Development Cost per Ounce: Blueberry and Scottie Gold Mine underground deposits have relatively high development costs per ounce of mineral resource. Expanding the resource for these areas would spread the relatively high development capital over more ounces, improving economics and reducing the AISC per ounce.

26.5 Metallurgy and Mineral Processing

Further metallurgical tests should be conducted to optimize processing flowsheet and improve gold recovery, including:

- Conduct systematic mineralogical investigations on the head samples, various middling streams and flotation tailings or leach residues to determine gold occurrences. With this knowledge, it may be possible to further improve gold recovery.
- Further conduct flotation and gravity concentration tests to verify and improve the gold recovery. The test outcomes should be able to optimize the process flowsheet.
- Conduct further ore-sorting tests on representative samples generated from various deposits to optimize the separation efficiency and improve concentrate grades. How to upgrade the fines that are not suitable for upgrading by ore-sorting should be investigated.
- Conduct a systematic test program to optimize cyanidation process flowsheet and operation parameters, especially the mineralization is expected to be treated by a toll milling plant which uses cyanide leach process. This test program should include grind size and cyanide dosage optimization. Residue cyanide detoxification conditions should be tested. The process flowsheet used at the toll milling plant should be thoroughly reviewed during the testing. The tests should also include gravity + cyanide leach integrated process.
- The combined process with incorporating ore-sorting preconcentration into flotation or cyanidation process should be investigated in an effort to improve project economics.
- Waste storge related geotechnical data and geochemical properties should be determined to better handle the waste materials.
- Some of design related data, such as settling rates, filtration rate, slurry viscosity, mineralization hardness and specific gravity, concentrate shipping related safety data (if concentrates will be one of the final products) should be further determined and confirmed.

There exists an opportunity for throughput expansion: The mine plan for the PEA is based on a nominal 1,000 tpd throughput scenario, which results in a 7-year mine life. Expanded resources are expected to have the potential to justify increased mine and mill throughput. As part of the upcoming Feasibility Study (FS), Scottie will evaluate the potential costs to expand the process plant capacity to 1,500-2,000 tpd with potential benefits to unit cost reductions for processing and G&A with respect to economies of scale.

For the toll milling option discussed in Section 24, there are additional opportunities of refinement on this concept include: (1) optimizing mine plan and resource for improving project economics (i.e., include more lower grade ounces), (2) elimination of the crushing/ore-sorting plant, and (3) further metallurgical test work on the mineralization to maximize recovery in a toll milling scenario. In the PEA, the cyanide leaching recoveries is set at 89.1% for the toll milling option. Historic production and recent test work suggests potential for improved gold recoveries to approximately 91% to 95%. The results from the preliminary metallurgical test work also suggested that a finer grinding would increase leaching recoveries of up to 97% for both gold and gold gravity concentrates. Scottie intends to follow up these promising results with further test work to be completed and incorporated into the Feasibility Study (FS).

26.6 Infrastructure

There are opportunities to optimize the construction timeline further, such as early mobilization and modularization of the process plant, that should be investigated during next phase of the study.

Benefited by the proximity of the Scottie Gold Mine to the deep water port in Stewart, BC, sourcing more competitively priced equipment and supplies from Asia will be investigated, which is expected to help reducing the project capital costs. Fabrication, modularization and pre-commissioning of equipment and structures can be completed in Asia before being shipped to the Project site. This will also help reducing the initial capital cost for the Project.

Investigation and possible adaptation of newest building technologies for enhancing the energy efficiency of buildings and mechanical equipment should also be conducted during the next phase of the study.

26.7 Environmental

The following works are recommended: baseline initiation, hydrology monitoring program. hydrological investigation, geochemical characterization, site geotechnical and ground condition studies, continuous weather monitoring and a manage plan for mapping the path forward in conducting assessments and permitting in the future.

26.8 Implementation Cost

Table 26-1 presents the estimated cost to implement the suggested recommendations above.

Table 26-1: Estimated Cost to Implement Suggested Recommendations, Summary

Area	Amount ($ million)
Geology and Mineral Resources	18.0
Geotechnical and Hydrogeological	0.5
Mineral Processing and Metallurgical Testing*	1
Mining (including underground survey)	1
Process	0.2
Infrastructure	0.3
Environmental	5
Total	**26.0**

*Excluding metallurgical sample collection site work and test sample shipping

27.0 REFERENCES

ABH Engineering, 2025. Phase I Particle Sorting Study – Engineering Report pp 63.

Alldrick, D.J., 1993. Geology and Metallogeny of the Stewart Mining Camp, Northwestern British Columbia. British Columbia Geological Survey Bulletin 85.

Alldrick, D.J., Höy, T., 1997. Intrusion-related gold-pyrrhotite veins. The Gangue 55, 8–10.

ALS Kamloops, 2025. Metallurgical Test Data Reports (Crushing and Grinding Bond Work Index and Abrasion Work Index), Project Number: KM7730.

Ascot Resources, 2020. Premier & Red Mountain Gold Project Feasibility Study NI 43-101 Technical Report https://ascotgold.com/projects/resources/.

Ascot Resources, 2022. NI 43-101 Resources Updated. Project Resources on https://ascotgold.com/projects/resources/.

BCMEM, 2018a. Hollywood Record Summary, Minfile No. 104B037. BCMEM, 2018b. St. Eugene Record Summary, Minfile No. 104B036.

BCMEM, 2017. East Gold Record Summary, Minfile No. 104B 033. BCMEM,2012. Hidden Creek Record Summary No. 103P 021.

BCMEM, 2017. East Gold Record Summary, Minfile No. 104B 033. BCMEM,2012. Hidden Creek Record Summary No. 103P 021.

Branson, T., Campbell, C., Voordouw, R., Mumford, T., Guestrin, D., Letham, E., 2020. 2019 Drilling, Geological, and Geochemical Report on the Scottie Gold Mine Project, BCMEM Assessment Report No. 39093.

Breitsprecher, K., Mortensen, J.K., 2004, BC Age 2004A-1: A database of Isotopic Age Determinations for Rock Units from British Columbia, BCGS Open File 2004-03.

Canadian Institute of Mining, Metallurgy and Petroleum (2014): CIM DEFINITION STANDARDS – For Mineral Resources and Mineral Reserves, CIM Standing Committee on Reserve Definitions, adopted May 10, 2014. Coffin, D., 1988. Preliminary Evaluation of the Hollywood Property, BCMEM Assessment Report No. 18201.

Clary, 1992. Metallurgical Scoping Testwork – Premier Gold Project Metallurgical Department.

Cutts, J.A., McNicoll, V.J., Zagorevski, A., Anderson, R.G., Martin, K., 2015. U-Pb geochronology of the Hazelton Group in the McTagg anticlinorium, Iskut River area, northwestern British Columbia, in: Geological Fieldwork 2014. British Columbia Geological Survey Paper 2015-1, pp. 87–101.

Decade Resources, 2014. New Discovery of Big M Vein yields up to 3418.07 g/t Gold in Trench Sampling, News release dated 23 June 2014.

DeLeen, J., 1980. 1980 Trenching and Sampling Undertaken on the Silver Bar Property, BCMEM Assessment Report No. 08909.

Dick, D.L., 1987. F.A.M.E. report, Summit Lake mine, Royal Scot Resources Ltd., July to October 1987, BCMEM Assessment Report No. 16768.

Fraser, S., Merber, D., Baria, O., 1983. Geology and epithermal gold mineralization within the Tide Lake Mineral Reserve, Internal report SJV83-330.10 for Summit Joint Venture.

Guestrin, D., 2019. 2019 Geological and Geochemical Report on the Stock Property, BCMEM Assessment Report No. 38615.

Guestrin, D., 2020. 2019 Geological and Geochemical report on the Summit Lake Property, BCMEM Assessment Report No. 39023.

Guestrin, D., and Nuttall, D., 2022. 2021 Drilling and Geochemical Report on the Scottie Gold Mine Property. BCMEM Assessment Report.

Gunning, D.R., Visagie, D.A., 2006. Technical report for the Summit Lake property, NI 43-101 technical report for Tenajon Resources Corp.

Higgs, A.A., Giroux, G., 2015. 2015 technical report for the Dolly Varden property, NI 43-101 technical report prepared for Dolly Varden Silver Corp.

Jones, I.W.O., 2013. Pretium Resources Inc.: Brucejack Project mineral resources update technical report, NI 43-101 report prepared for Pretium Resources Inc.

Kikauka, A., 2016. Geochemical & petrographic report. Summit Lake project, Summit Lake, Stewart area, British Columbia, Canada, BCMEM Assessment Report No. 35805.

Kikauka, A., 2012. Summit 5 & 6 (583912 & 993684) Project – Geological, Geochemical and Geophysical Report, BCMEM Assessment Report No. 33386.

Kikauka, A., 2007. Geochemical Report on the Summit Lake Group of Claims, BCMEM Assessment Report No.28918.

Kikauka, A., 2004. Geophysical and geochemical report on the Summit 5 (mineral tenure), Summit Lake, Stewart, BC, BCMEM Assessment Report No. 27448.

Kikauka, A., 2003. Geological and geochemical report on the Summit 5 claim, Summit Lake, Stewart, BC, BCMEM Assessment Report No. 27163.

Kikauka, A., 2000. Geological, geophysical and geochemical report on the Summit claims, Summit Lake, Stewart, BC, BCMEM Assessment Report No. 26378.

Kikauka, A., 1998. Geological, Geophysical, and Geochemical Report on the Summit Claims, BCMEM Assessment Report No. 25677.

Kikauka, A., 1996. Geological and Geochemical Report on the Summit Claims, BCMEM Assessment Report No.24912.

Kretschmar, U., Kretschmar, D., 1980. Geological Report – Hollywood Claim Group, BCMEM Assessment Report No. 08520.

Kruchkowski, E.R., 2017. Assessment report on Scottie Gold property, BCMEM Assessment Report No. 36674.

LeFort, D., Ibbotson, M., and Nuttall, D., 2023. 2022 Drilling and Geochemical Report on the Scottie Gold Mine Property. BCMEM Assessment Report.

Lefort, D., McGillivray, C., 2024. 2023 Drilling and Geochemistry Report on the Scottie Gold Mine and the Tide North Properties. BCMEM Assessment Report.

Lewis, P.D., 2013. Iskut River area geology NTS sheets 104B/08, 09, 10 and part of 104B/01, 07, 11. Geoscience BC report 2013-05.

McCormack, J., 1985. Status of Operations as of Jan 31, 1985, Internal report for Scottie Gold Mines Ltd. McGuigan, P.J., Wilson, L., 1985. 1984 field exploration program and diamond drilling program on the Summit group claims, BCMEM Assessment Report No. 13593.

Monahan, M., 1983. The Hollywood Prospect 1983 Exploration, BCMEM Assessment Report No. 11987. Monger, J.W., 1984. Cordilleran tectonics ; a Canadian perspective. Bulletin de la Société Géologique de France 7, 255–278.

Morrison, R., McKinnon, C., Liukko, G., Kesavanathan, D., Gagnon, A., Hafez, S.A., Danon-Schaffer, M., McLaughlin, M., Ouellet, J., 2013. Preliminary economic assessment for the Granduc copper project, northern British Columbia, NI 43-101 report for Castle Resources Inc.

Mumford T., 2025. Letter from Scottie Resources Corp. to Moose Mountain Technical Services on Scottie gold Property claims and royalties.

Nelson, J., Waldron, J., van Straaten, B., Zagorevski, A., Rees, C., 2018. Revised stratigraphy of the Hazelton Group in the Iskut River region, northwestern British Columbia, in: Geological Fieldwork 2017, British Columbia Geological Survey Paper 2018-1. British Columbia Ministry of Energy, Mines and Petroleum Resources, pp. 15–38.

Nuttall, D.R., Guestrin, D., Branson, T., Mumford, T., Voordouw, R., 2021. 2020 Drilling, Geological, Geochemicaland Geophysical Report on the Scottie Gold Mine Property, BCMEM Assessment Report No. 39397.

Petersen, D.B., Vulimiri, M.R., 1989. Royal Scot Resources Ltd. Summit Lake Property Summary Report and Exploration Proposal June 1989 (Internal Company Report).

Poon, J., 2016. Airborne Geophysical Survey Report: Summit Lake Block, BCMEM Assessment Report No. 36318.

Pretium Resources, 2019. Brucejack project reserves and resources update, News release dated April 2019.Pretium Resources, 2011. Snowfield resources increase. News release 23 February 2011.

Ramsay, A., 2013. 2012 airborne magnetic geophysical survey on the Summit Lake property, northwestern British Columbia, BCMEM Assessment Report No. 33511.

Rhys, D., 2006. Summary of August 2006 structural work at the Summit Lake gold mine, Unpublished report for Tenajon Resources.

Rhys, D.A., 1995. The Red Bluff Gold-Copper Porphyry and associated Precious and Base Metal Veins, Northwestern British Columbia, in: Tom G. Schroeter (Ed.), Porphyry Deposits of the Northwestern Cordillera of North America. CIM Special Volume 46, pp. 838–850.

Rhys, D.A., Sieb, M., Frostad, S.R., Swanson, C.L., Prefontaine, M.A., Mortensen, J.K., Smit, H.Q., 1995. Geology and setting of the Red Mountain gold–silver deposits, northwestern British Columbia, in: Tom G. Schroeter (Ed.),Porphyry Deposits of the Northwestern Cordillera of North America. CIM, pp. 811– 828. Roth, T., Thompson, J.F.H., Barrett, T.J., 1997. The Precious metal-rich Eskay Creek deposit, Northwestern British Columbia, in: Volcanic-Associated Massive Sulfide Deposits: Processes and Examples in Modern and Ancient Settings, Reviews in Economic Geology. Society of Economic Geologists, pp. 357– 373.

Scottie Resources, 2019a. Rotation to change name to Scottie Resources Corp, News release dated 10 January 2019.

Scottie Resources, 2019b. Scottie acquires option for 100% interest in the Bow property, News release dated 25 January 2019.

Scottie Resources, 2017. Rotation completes purchase of Scottie Gold property; News release dated 13 December 2017.

Seabridge Gold, 2024. https://www.seabridgegold.com/press-release/seabridge-gold-reports-bronson- slope-resource-estimate-at-its-iskut-project-in-bc

Sepro, 2023. Serac Exploration – Metallurgical Test Work Report, Project Number:MS2109, pp94.

Sepro 2025. Scottie Resources – Heavy Liquid Separation Report, Project Number: MS2199, pp34.

Sepro, 2025. Metallurgical Test Data Reports (Head Assay and Determinations of Moisture, Ball Mill Work Index, Slope of Repose, Bulk Density and Specific Gravity), Project Number: MS 2209.

Sepro, 2025. Metallurgical Test Data Reports (Size by Size Assay), Project Number: MS 1563.Seraphim, R.H., 1947. The Scottie Group; Bachelor Thesis, University of British Columbia. Snip Gold Corporation, 2022. Bronson Slope Deposit. FAQs on iskutproject.com.

Sheldrake, R.F., 1983. Report on a multi-frequency electromagnetic and magnetic survey in the Summit Lake area, BCMEM Assessment Report No. 12342.

Sinclair, A.J., 1981. Mineralographic study of massive sulphide chips, pp4.

Skeena Resources, 2022. Snip. Project overview on www.skeenaresources.com. Skeena Resources, 2022. Eskay Creek. Project overview on www.skeenaresources.com.

Skeena Resources, 2023. Eskay Creek. Press Release: Skeena files updated feasibility Study Technical Report for Eskay Creek.

Stanley, B., and Nelson, J., 2022. Revised stratigraphy of the Stuhini and Hazelton groups and LA-ICP-MS zircon geochronology of the Scottie gold mine area, northwestern British Columbia. In: Geological Fieldwork 2021, British Columbia Ministry of Energy, Mines and Low Carbon Innovation, British Columbia Geological Survey Paper 2022-01, pp. 83-102.

Stevens, G., 2006. Outland Silver-Prospecting Report, BCMEM Assessment Report No. 28070.

Strickland, D., 2018. Assessment Report on the Stock Property, BCMEM Assessment Report No. 37446. Tribe, N.L., Gewargis, W., Hicks, K., 1983. Report on 1983 Field Exploration Program and Underground Drilling Program, Unpublished Report for Scottie Gold Mines Ltd.

Tribe, N.L., 1985. Report on the 1984 Field Exploration Program, Unpublished Report for Scottie Gold Mines Ltd.Tudor Gold, 2022. Treaty Creek. Project overview on www.tudor-gold.com/projects/treaty- creek/.

Tudor Gold Assessment Report. April 5, 2024 – https://wp-tudorgold-2023.s3.ca-central-1.amazonaws.com/wp-content/uploads/2024/04/Tudor-Treaty-Creek-NI-43-101-Technical-Report-Update-FINAL.pdf

Turner, A.J., Nicholls, S.J., 2019. Technical Report and Mineral Resource Update for the Dolly Varden Property, British Columbia, Canada (NI 43-101 Technical Report).

Visagie, D.A., 1989. Summit Property Geochemical Report, BCMEM Assessment Report No. 19302.

Visagie, D.A., Varas, P., 1991. 1990 Summary Report, Summit Lake property, BCMEM Assessment Report No. 20987.

Visagie, D.A., 1995. Geochemical report, RS#1 group, BCMEM Assessment Report No. 23874.

Visagie, D.A., 2004. Technical report for the Summit Lake property, NI 43-101 report for Tenajon Resources Corp.

Voordouw, R., Carr, I., 2019. 2018 Geological and Geochemical Report on the Scottie Gold Mine Project, BCMEM Assessment Report No. 38246.

Westmin Resources, 1994. Mineralogical examination of mill-processed samples from the Scottie Gold Mine, pp. 12.

28.0 QP CERTIFICATES

I, Sue Bird, P.Eng., am employed as a Geological Engineer with Moose Mountain Technical Services, with an office address of #210 1510 2nd Street North Cranbrook, BC V1C 3L2. This certificate applies to the technical report titled “Preliminary Economic Assessment, NI 43-101 Technical Report, for the Scottie Gold Mine Project In Northwestern British Columbia, Canada”, with an effective date of ctober 28, 2025 (the “Technical Report”).

- I am a member of the self-regulating Association of Professional Engineers and Geoscientists of British Columbia (#25007). I graduated with a Geologic Engineering degree (B.Sc.) from the Queen’s n iversity in 1989 and a M.Sc. in Mining from Queen’s ni versity in 1993.
- I have worked as an engineering geologist for over 25 years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations and evaluations. Similar resource estimate projects specifically include those done for rte mis’ Blackwater gold project, scot ’s Premier Gold Project, Spanish Mountain Gold, all in BC; KSM’s Courageous Lake deposit in NWT, 3 ’s Marban and Garrison, gold projects in Quebec and Ontario, respectively, as well as numerous due diligence gold projects in the southern US done confidentially for various clients.
- As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).
- I visited the Scottie Gold Mine Project site on August 6, 2025, for one day.
- I am responsible for Sections 1.1 through 1.8, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0 (except 12.5.2, 12.5.3 and 12.5.4), 14.0, 23.0, 25.2, 26.2 and 27.0 (only references from sections for which I am responsible).
- I am independent of Scottie Resources Corp. as independence is described by Section 1.5 of NI 43–101.
- I have had prior involvement with the Scottie Gold Mine property that is the subject of the Technical Report, in acting as a Qualified Person for the technical report titled “NI 43-101 2025 Maiden Mineral Resource Estimate for the Scottie Gold Mine Project” that has an effective date of eb ruary 2, 2025.
- I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
- As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 28th day of October, 2025

“signed and sealed”

Sue Bird, P.Eng.
Geological Engineer
Moose Mountain Technical Services

I, Hassan Ghaffari, P.Eng., M.A.Sc., do hereby certify that:

- I am a Director of Metallurgy with Tetra Tech Canada Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report titled "Preliminary Economic Assessment, NI 43-101 Technical Report, for the Scottie Gold Mine Project In Northwestern British Columbia, Canada", with an effective date of October 28, 2025 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#30408).
- My relevant experience includes more than 30 years of experience in mining and mineral processing plant operation, engineering, project studies and management of various types of mineral processing, including hydrometallurgical processing for porphyry mineral deposits.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I conducted a personal inspection of the Scottie Gold Mine property on July 29, 2025 and inspected the overall project site and the proposed infrastructure areas.
- I am responsible for Sections 1.12 through 1.16, 2.0, 3.0, 12.5.3, 18.0, 20.0, 21.1, 21.2, 22.0, 24.0, 25.1, 25.5 to 25.7, 26.1, 26.3, 26.6 to 26.8 and 27.0 (only references from sections for which I am responsible).
- I am independent of Scottie Resources Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have had no prior involvement with the Scottie Gold Mine property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 28th day of October, 2025

"signed and sealed"

Hassan Ghaffari, P.Eng., M.A.Sc.
Director of Metallurgy
Tetra Tech Canada Inc.

I, Jianhui (John) Huang, Ph.D., P.Eng., do hereby certify that:

- I am a Senior Metallurgist with Tetra Tech Canada Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report titled “Preliminary Economic s sessment, NI 43-101 Technical Report, for the Scottie Gold Mine Project In Northwestern British Columbia, Canada”, with an effective date of ctober 28, 2025 (the “Technical Report”).
- I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#30898).
- My relevant experience includes over 40 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores, and mineral processing plant operation and engineering including hydrometallurgical mineral processing for porphyry mineral deposits.
- I am a “Qualified Person” for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I have not conducted a personal inspection of the Scottie Gold Mine property. I visited Sepro metallurgical laboratory on August 27, 2025 to inspect the sample tested and the laboratory and witnessed the DMS test sample preparation.
- I am responsible for Sections 1.10, 1.11, 12.5.4, 13.0, 17.0, 19.0, 21.3 (excluding 21.3.2), 25.4, 26.5 and 27.0 (only references from sections for which I am responsible).
- I am independent of Scottie Resources Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have had no prior involvement with the Scottie Gold Mine property that is the subject of the Technical Report.
- I have read NI 43-101 Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 28th day of October, 2025

“signed and sealed”

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech Canada Inc.

I, Damian Gregory, P.Eng., do hereby certify that:

- I am a Principal Consultant at Datamine Canada Inc. (Snowden Optiro) 5-1760 Regent Street , Sudbury, Ontario, P3E 3Z8, CANADA contact@snowdenoptiro.com www.snowdenoptiro.com
- This certificate applies to the technical report titled "Preliminary Economic s sessment, NI 43-101 Technical Report, for the Scottie Gold Mine Project In Northwestern British Columbia, Canada", with an effective date of ctober 28, 2025 (the "Technical Report").
- I am a "Qualified Person" for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing. My qualifications as a qualified person are as follows:
  - I am a master's graduate of Laurentian n iversity (2007) and have a bachelor's degree from University of Mining and Geology, Bulgaria (1995)
  - I am a professional engineer registered in Ontario (License Number: 100107186)
  - My relevant mining experience after graduation is over 25 years. My consulting experience with open pit optimization is 15 years.
- I conducted a personal inspection of the Scottie Gold Mine property on July 29, 2025 and inspected the overall project site and the proposed mining areas.
- I am responsible for Sections 1.9, 12.5.2, 16.0, 21.3.2, 25.3, 26.4 and 27.0 (only references from sections for which I am responsible).
- I am independent of Scottie Resources Corp. as Independence is defined by Section 1.5 of NI 43-101.
- I have had no prior involvement with the Scottie Gold Mine property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 28th day of October, 2025

"signed and sealed"

Damian Gregory, P.Eng
Principal Consultant, Snowden Optiro