HARTE GOLD CORP



NI 43-101 TECHNICAL REPORT

Feasibility Study for Expansion to 1,200 t/d

for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario

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The undersigned prepared this Technical Report, titled *National Instrument 43-101 Technical Report Feasibility Study for Expansion to 1,200 t/d, for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario, Canada*, and dated *5 March 2021*, in support of the public disclosure for public listing. The format and content of this report conforms to National Instrument 43-101 (NI 43-101) of the Canadian Securities Administrators.

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GLOSSARY

Units of Measure

a	year
amsl	above mean sea level
asl	above sea level
cm	centimetre(s)
°C	degree Celsius
ft	foot
g/t	grams per tonne
Ga	billion years
ha	hectare
k	thousand
km	kilometre
kW	kilowatt
L	litre
L/s	litres per second
lb	pound
m*g/t	Grade Thickness – metres multiplied by gold grade in g/t
m/s	metres per second

m	metre
m^3	cubic metre
Ma	millions of years
masl	metres above sea level
mbs	metres below surface
mm	millimetre
Moz	million troy ounces
m RL	metres relative level
Mt	mega tonne or million tonnes
MW	megawatts
µS/cm	microsiemens per centimetre
oz	troy ounce
ppm	parts per million
t	tonne
T	short ton
t/d	tonnes per day
wmt	wet metric tonnes
yd	yard

Abbreviations and Acronyms

AAS	atomic absorption spectrometry
Access Agreement	agreement with the remote lodge operator
Actlabs	Activation Laboratories
Ag	silver
AGAT	AGAT Laboratories
Ai	Bond abrasion index
ALS	ALS Minerals Ltd.
AMEC FW	AMEC Foster Wheeler (now Wood Group)
ALS	ALS Minerals Ltd.
Appian	ANR Investments B.V.
Asahi	Asahi Refining Canada Ltd.
Au	gold
A x b	resistance to impact breakage
Belt	Dayohessarah Greenstone Belt
BGM	bituminous heomembrane liner
BM1 and BM2	ball mill 1 and ball mill 2
BMF	ball mill feed
BQE	BQE Water
BWi	Bond ball mill work index
$	Canadian Dollar

$M	dollars, millions
CaO	calcium oxide
CAPEX	capital and sustaining capital expenditures
CSS	closed side set
CDA	Canadian Dam Association
CIL	carbon in leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	*CIM Definition Standards for Mineral Resources & Mineral Reserves*
CIP	carbon in pulp
CMS	cavity monitoring system
CN	cyanide
CN_{WAD}	weak acid dissociable cyanide
CND	dissociable cyanide
CN_T	total cyanide
Company	Harte Gold
conc	concentrate
Corona	Corona Gold Corporation
Corona Option	Harte Gold and Corona Option Agreement
CRM	certified reference material
Cu	copper
CWi	Bond impact crushability work index
DA	dynamic anisotropy
DO	dissolved oxygen
DSO	Deswik Stope Optimizer
DST	DST Consulting Engineers Inc
EA	environmental assessment
EAG	Environmental Applications Group
EDF	Environmental Design Flood
ELOS	equivalent linear overbreak and sloughage
EM	electromagnetic
EMS	environmental management system
Escan	3-D resistivity survey
ESDM	emission summary and dispersion modelling
ESS	electrical substation
ETP	effluent treatment plant
FA	fresh air
FAR	fresh-air raise
Feasibility Study	Feasibility Study for Expansion to 1,200 t/d
FT	flotation tailings
FW	footwall

FWC	full-width composites
G&A	general and administrative
GISTM	*Global Industry Standard on Tailings Management*
Glencore	Glencore Canada Corporation
Halverson option	Lloyd Halverson, Doug Kakeeway, and John E. Ternowesky Property Option Agreement
Halyard	Halyard Inc.
HARD	half absolute relative difference
Harte Gold	Harte Gold Corp.
Hemlo Gold	Hemlo Gold Mines Inc.
HDPE	high-density polyethylene
HPC	Hazard Potential Classification
HR	hydraulic radius
HW	hanging wall
Hydro One	Hydro One Networks, Inc.
IBA	Impact Benefits Agreement
ID, ID2, ID3	inverse distance, inverse distance squared, inverse distance cubed
IDF	Inflow Design Flood
IP	induced polarization
IRR	internal rate of return
ISO	International Organization for Standardization
JMVL	Jodouin Mine Ventilation Ltd.
jumbo	electric-hydraulic powered development drill jumbo, typically with one or two drill booms
JV	joint venture
LBMA	London Bullion Metals Association
level	mine working level referring to the nominal elevation (m RL), e.g., 4285 level (mine workings at 4285 m RL)
LHD	load, haul, and dump unit (underground loader or scooptram)
lidar	Light Detection and Ranging
LOM	life of mine
LRIA	*Lakes and Rivers Improvement Act*
Mag	magnetic
Maximos	Maximos Metals Corp.
MC	Merrill-Crowe
MECP	Ontario Ministry of the Environment, Conservation and Parks
MENDM	Ontario Ministry of Energy, Northern Development and Mines
Met-Solve	Met-Solve Laboratories Inc.
MIBC	methyl isobutyl carbinol
MNDM	Ontario Ministry of Northern Development and Mines
MOE	Ministry of Environment
MPC	mine power centre

NaCN	sodium cyanide
NAD	North American Datum
NE	northeast
NI	National Instrument
Ni	nickel
NN	nearest neighbour
Non-PAG	non-potentially acid generating
NPI	Net Profits Interest
NPV	net present value
$NPV_{5\%}$	net present value at a 5% discount rate
NSR	net smelter royalty
NTMF	north tailings management facility
NW	northwest
OK	ordinary kriging
OMC	optimum moisture content
Operation	Sugar Zone Mine, including the active underground workings, surface fixed plant, mineral processing plant, and future south tailings management facility
OPEX	operating cost
Orion	Orion Mine Finance
P_{80}	80% passing
P&C	Paterson & Cooke
P&E	P&E Mining Consultants Inc.
P.Eng.	Professional Engineer
P.Geo.	Professional Geoscientist
PAX	potassium amyl xanthate
Pb	lead
PEA	Preliminary Economic Assessment
PGA	peak horizontal ground acceleration
PLS	pregnant leach solution
PMF	Probable Maximum Flood
PMFN	Pic Mobert First Nation
Project, the	increase throughput from 800 t/d to 1,200 t/d
Property, the	the Sugar Zone property, consisting of 80,000 ha of mining claims and leases
Q1, Q2, Q3, Q4	first quarter, second quarter, third quarter, fourth quarter of the year
QA/QC	quality assurance/quality control
QEM-ARMS	automated rapid mineral scan
QP	Qualified Person
QTO	quantity take-off
RA	return air
RAR	return-air raise

RockEng	RockEng Inc. (previously Mine Design Engineering Inc.)
ROM	run of mine
RWi	Bond rod mill work index
S	sulphur
SAG	semi-autogenous grinding
SCC	Standards Council of Canada
SDZ	Sugar Deformation Zone
SFL	Sustainable Forest License
SE	southeast
SGS	SGS Canada Inc.
SMC	SAG mill comminution
SPMDD	standard Proctor maximum dry density
SPT	Standard Penetration Test
Standard	Certified reference material
Stantec	Stantec Consulting Services Inc.
STMF	south tailings management facility
SW	southwest
t_a	resistance to abrasion breakage
Technical Report	the present report, *National Instrument 43-101 Technical Report Feasibility Study for Expansion to 1,200 t/d, Harte Gold Sugar Zone Mine, Ontario, Canada*
TMF	tailings management facility
UDS	underground (paste) distribution system
UG	underground
UPS	uninterruptible power supply
US$	United States dollar(s)
UTM	Universal Transverse Mercator grid system
VFD	variable-frequency drive
VTEM plus	versatile time-domain electromagnetic
VLF	very-low frequency
VSD	variable-speed drive
Wesdome	Eagle River Mine
WGM	Watts, Griffis and McOuat Limited
Wi	Bond work index
Wood	Wood Plc
WRB	waste rock barricade
XRD	X-ray diffraction
Zn	zinc

1 SUMMARY

In a press release dated 20 January 2021, Harte Gold Corp. (Harte Gold or Company) announced the results of its Feasibility Study for Expansion to 1,200 t/d (Feasibility Study) on the Sugar Zone Mine (Operation), located in Northern Ontario, Canada. Having successfully entered commercial production in Q1 2019 with a planned production rate of 800 t/d, the objective of the Feasibility Study was to consider the most cost- and capital-effective strategy to increase annual production to 1,200 t/d (the Project). This National Instrument (NI) Technical Report—Feasibility Study for Expansion to 1,200t/d Harte Gold Sugar Zone Mine, Ontario, Canada (the Technical Report) also supports the disclosure of an updated Mineral Resource Estimate (effective date 30 September 2020) and an updated Mineral Reserves Estimate (effective date 31 December 2020). This Technical Report outlines the results of the Feasibility Study and was prepared in accordance with the National Instrument 43-101 (NI 43-101) of the Canadian Securities Administrators.

The Operation has been significantly de-risked through almost 9,000 m of underground pre-production development, as well as over 430,000 tonnes of ore mined and processed through a combination of longhole stoping and underground sill development to recover over 70,000 oz of gold through the end of 2020. Mining began with an advanced exploration bulk sample program that recovered over 70,000 tonnes of ore between 2016 and 2017, followed by pre-commercial production mining of 47,000 tonnes of ore between 2017 and 2018. Since the declaration of Commercial Production on 1 January 2019, the Company has mined 313,000 tonnes of ore from the South and North Zones of the Sugar Zone orebody and has significantly advanced development toward the Middle Zone orebody.

Harte Gold used the services of several consulting firms to design and cost the components making up the key elements of the Project. They include Halyard Inc. (Halyard), Paterson & Cooke Canada Inc. (P&C), Deswik Consulting (Deswik), Wood Canada Limited (Wood), Jodouin Mine Ventilation Ltd. (JMVL), Environmental Applications Group (EAG), BQE Water (BQW), Stantec Consulting Services Inc. (Stantec), and Ausenco Engineering Canada (Ausenco).

All costs are in Canadian dollars, unless otherwise stated.

This Technical Report uses the International System of Units (SI) (metric), except for troy ounces, used to report the mass of precious metals, and some imperial measures that remain in common use.

1.1 Property Description and Location

The Sugar Zone Mine and its surrounding mining claims and leases (Property) are located in northern Ontario within the Sault Ste. Marie Mining Division. The Property is located 30 km north of White River, which is on Ontario Highway 17 (part of the Trans-Canada Highway system), approximately midway between Thunder Bay and Sault Ste. Marie. A Property location map is presented in Figure 1-1.

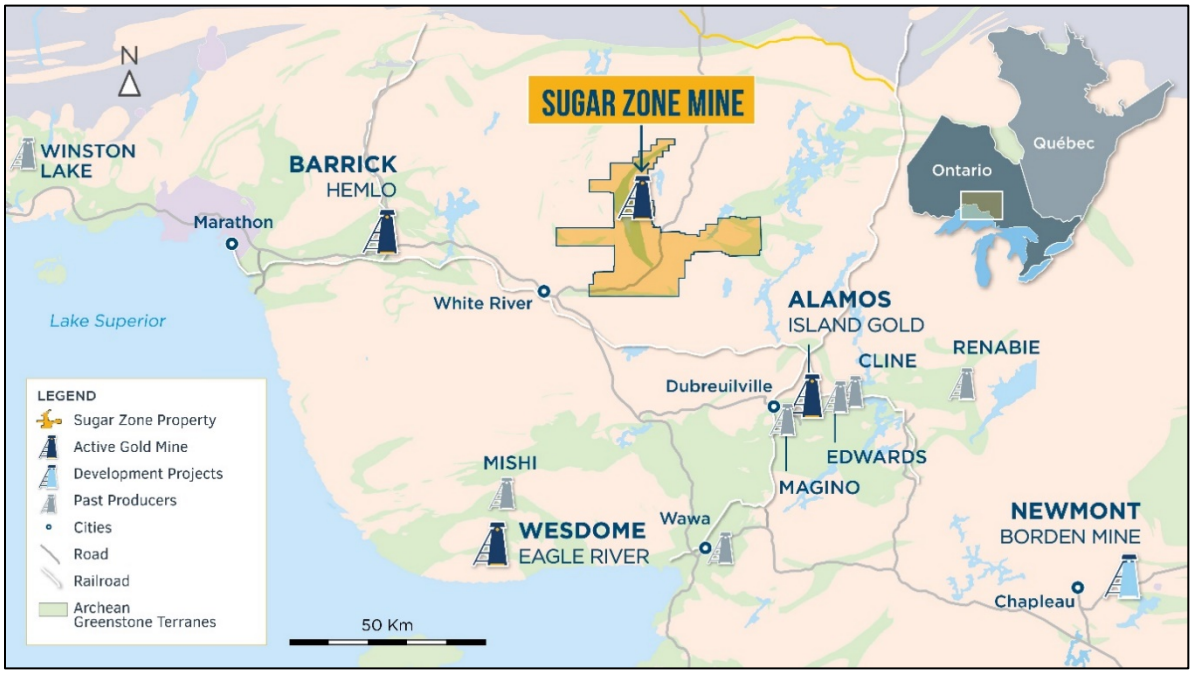


Figure 1-1: Location Map of Harte Gold Property

The Property comprises 81,287 ha and consists of 69 boundary cell claims, 43 single cell claims, 197 multi-cell claims and four mining leases (1,467.26 ha) that are 100% owned by Harte Gold Corp.

The Property is subject to the following net smelter royalties (NSR):

- 2.0% NSR held by Lloyd Halverson, John E. Ternowesky, Ernie Beaven, Eino Ranta, the Estate of Omer L. Belisle, Broad Horizons Trust, and Broad Horizons Inc., granted on 10 July 1998 (Corona/Harte JV Royalty).
- 1.5% NSR held by 2729992 Ontario Corp. (an affiliate of ANR Investments B.V.) granted on 18 December 2019 (Appian 2019 Royalty).
- 0.5% NSR held by 2729992 Ontario Corp. (an affiliate of ANR Investments B.V.) granted on 17 July 2020(Appian 2020 Royalty).

1.2 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The Operation is accessible by road from Highway 631, a secondary paved highway extending north from White River, which passes approximately 10 km east of the Property. A Hydro One electrical transmission line and the Canadian Pacific transcontinental main rail line pass through White River.

The Operation benefits significantly from proximity to the Hemlo Gold Camp and several mining operations in the Wawa area; a number of businesses in the area provide services to the exploration and mining sector, and a long history of mining in the region contributes to a highly skilled work force.

The Property lies in the Great Lakes watershed. The topography of the area is typical of the Canadian Shield and consists of a peneplained surface, with local relief consisting of low rocky ridges separated by poorly drained ground. The area around the Property has local relief varying from 394 masl at Dayohessarah Lake to 480 masl on higher ridges. The Property area is characterized by a boreal climate, with short, hot summers and cold, snowy winters. Snow cover occurs from late October through April. Exploration and mining activities are generally carried out year-round.

1.3 History

The Property has a history of significant exploration that has been undertaken intermittently since 1969. The majority of the work has been done in the vicinity of Dayohessarah Lake and the Sugar Zone area and follows the discovery of the Hemlo Gold Deposit in 1981. Hemlo Gold Mines Inc. discovered the Sugar Zone deposit by surface prospecting in 1991, followed by trenching and diamond drilling between 1991 and 1994. The Property was explored by the Corona Gold Corporation/Harte Gold Corp. Joint Venture between 1998 and 2009, and by Harte Gold after 2010. To date, there have been 972 holes drilled on the Property, for a total of 355,593 m.

Harte Gold completed a 70,000-tonne underground bulk sampling program on the Sugar Zone in 2017. The program included approximately 960 m of ramp development and 634 m of horizontal drifting in mineralization on five levels at a 15 m vertical spacing, to provide drilling access for longhole mining. A total of 15 stopes were mined from development work completed on the 375, 360, 345, 330, and 315 masl levels. An extensive sampling and grade-control program was completed, which included chip, miner muck, crusher muck, and truck muck. Toll processing of the bulk sample was completed at Barrick's Williams Mine process plant in the Hemlo camp.

All commercial production permits were issued in September 2018. Process plant construction and transition to electric grid power were completed in September 2018, with the first gold doré bar poured in October 2018. Process plant commissioning was completed in early November 2018. The Premier of Ontario and Minister of Energy, Northern Development and Mines attended an official mine opening on 24 October 2018. The first production blast was completed in the mine in early November 2018.

The underground mining contractor, Technica Mining, completed the bulk sample and some pre-production mining. Underground mining contractor Redpath did the remainder of pre-production work and all mining through August 2020. Between September 2020 and December 2020, Harte Gold recruited and hired its own underground workforce and successfully completed the transition to Owner mining prior to the end of 2020.

1.4 Geological Setting and Mineralization

The Property is in the Dayohessarah Greenstone Belt (the Belt) of late Archean (ca. 2.7 Ga) age that is part of the Abitibi-Wawa Subprovince of the Superior Province. The Belt is approximately 36 km long

and varies from 1.5 to 5.5 km wide. Principal lithologies in the Belt are moderately to highly deformed, metamorphosed volcanics, volcanoclastics, and sediments that have been enclosed and intruded by tonalitic to granodioritic quartz-porphyry plutons.

In the Sugar Zone gold deposit (Sugar Zone deposit), gold mineralization occurs in quartz veins, quartz stringers, and quartz-flooded zones predominantly associated with narrow porphyry sills, porphyry contact zones, hydrothermally altered mafic metavolcanics, and rarely in weakly altered mafic metavolcanics. The mineralization occurs in three parallel zones, the Upper, Lower, and Footwall Subzones, that range from 0.2 to 6 m thick, strike at 140°, and dip between −65° and −75° to the west. The three subzones are separated by 20 to 30 m of non-mineralized metavolcanics. The Sugar Deformation Zone (SDZ) is a high-strain deformation zone associated with the Sugar Zone mineralization, which has a northwest trend and dips west at 65° to 75°. The mineralization has been defined over a 3.5 km strike length and to a vertical depth of over 1,200 m, and remains open along strike and at depth. Fine- to coarse-grained specks and blebs of visible gold are common in the Sugar Zone quartz veins, usually occurring within marginal, laminated, or refractured portions of the veins. Quartz veins and silicified rocks also contain varying amounts of pyrrhotite, pyrite, chalcopyrite, galena, sphalerite, molybdenite, and arsenopyrite.

1.5 Deposit Type

The Sugar Zone deposit is interpreted as an orogenic, mesothermal gold deposit in a zone of high strain within the Belt. The deposit is hosted in medium-metamorphic grade (amphibolite) rocks that exhibit ductile deformation, and have been intruded by felsite and porphyry sills. The gold is associated with silica-sulphide-potassic alteration. These characteristics are shared with the Hemlo deposit, 68 km to the west.

1.6 Exploration

The initial Mineral Resource estimate of the Sugar Zone deposit was prepared in 1998 and contained an estimated 430,000 at 11.19 g/t Au. Successive exploration work and drilling campaigns between 2000 and 2010 led to an NI 43-101 Mineral Resource estimate of 1.12 Mt of indicated material at an average grade of 8.41 g/t Au, 0.42 Mt of inferred material, and an average grade of 7.13 g/t Au.

Since 2010 Harte Gold has undertaken multiple geophysical surveys using several techniques that have included magnetic, electromagnetic, versatile time-domain electromagnetic (VTEM), very low frequency (VLF), 3-D resistivity, and induced polarization. These geophysical approaches have been applied property wide through use of aerial survey, and in local areas of interest with ground-based activities.

The geophysical survey results have assisted Harte Gold in identifying prospective targets to focus on for more intensive prospecting, including grab samples, mechanical trenching, and diamond drilling. The major focus of the exploration programs between 2000 and 2010 focussed on defining and

expanding the Sugar Zone deposit and near-mine targets along strike, such as the Middle and Wolf Zones.

The work completed to date has led to the latest NI 43-101 Mineral Resource estimate of 2.8 Mt of material in the Indicated category at an average grade of 11.87 g/t Au, and 1.9 Mt of material in the Inferred category at an average grade of 9.45 g/t Au.

Exploration on the Property continues to be highly prospective, both near-mine and property-wide, and the Company is focussed on allocating resources to replace and grow the reserve base, as well as developing Property-wide targets.

1.7 Drilling

A total of 972 drill holes and 355,593 m of drilling have been completed on the Property since 1969. Hemlo Gold Mines initiated the first significant drill program on the Sugar Zone in 1993. The current Mineral Resource estimate is based on 750 drill holes for a total of 339,234 m that have been drilled since 1993.

The Sugar Zone has attracted the majority of the drilling programs since 2010, and until 2018 the focus was around increasing the size of and confidence in the Mineral Resource through drilling step-outs to extend the deposit both laterally along strike and vertically down dip. In 2019 the focus of Sugar Zone drilling shifted to the inferred Sugar South Extension that, when drilled at the appropriate spacing, could lead to additional mineral reserves near-surface, accessible by existing underground infrastructure.

In 2016 a geophysical anomaly was identified between the Sugar and Wolf Zones that led to the discovery of the Middle Zone. Through delineation drilling and infill campaigns, the Middle Zone was incorporated into the Mineral Reserves estimate.

In Q3 2020, a surface and underground definition drilling program was initiated to de-risk the short- to medium-term mine plan and to further delineate the lateral extents of the ore lenses. The results of the 2020 definition drilling have not been incorporated into the current Mineral Resource estimate.

Property-wide diamond drilling also occurred in 2020, focusing on prospective, near-surface targets identified through prospecting and geophysical survey techniques.

1.8 Sample Preparation, Analyses, and Security

Harte Gold drill-core sample intervals were sent for analysis to Actlabs in Ancaster, Ontario, for all assay work during the 2009 program and all subsequent programs. All samples were assayed for gold using a 30 g fire assay using lead collection and an AAS finish. The detection limit is 5 ppb Au. If sample assays returned a value greater than 3,000 ppb Au, they were re-assayed using a metallic screen

method. Actlabs is accredited by the Standards Council of Canada (SCC) for International Standards Organization (ISO/IEC) 17025, Mineral Analysis/geological tests (CAN-P-1579).

For Corona and Harte Gold's 2009 to 2020 programs, field-inserted certified reference standards and blanks supplemented Actlabs internal quality assurance/quality control (QA/QC) programs on blanks and standards. For the 2010 to 2019 programs, 30,381 samples were sent for analysis, including 1,660 blanks and 1,389 reference standards.

It is the QP's opinion that sample preparation, security, and analytical procedures for the Property's drill program were adequate for the purposes of this Mineral Resource estimate.

1.9 Data Verification

The QP's data verification has shown no significant issues with the integrity of historical data. The sampling techniques and data are considered to be of sufficient quality to carry out Mineral Resource estimations for the Sugar Zone Mine.

Visual validation of the drill-hole locations and mineralized intersections was undertaken and verified against hard-copy drill sections. Relative to each other and the cross-sections provided, the drill holes used as the basis for the Sugar Zone Mineral Resource estimates were considered acceptable for classification and reporting under NI 43-101 guidelines.

The QP has verified the data that underpin the Mineral Resource estimate contained in this Technical Report. The QP believes that data verification procedures undertaken adequately support the integrity of the data used in the Mineral Resource estimate.

1.10 Mineral Processing and Metallurgical Testing

The mill has been processing Sugar Zone ore since late 2018, with consistent recoveries of approximately 94%. The mill has an efficient gravity recovery circuit, and the Sugar Zone ore contains sufficient coarse gold to recover over 65% of the gold in the circuit to then be smelted into doré bars. This Feasibility Study considers that 62.5% of the gold will be recovered through the gravity circuit over the life-of-mine (LOM). The existing mill has a flotation circuit consisting of rougher and scavenger cells that typically produce a concentrate containing approximately 100 g/t Au, and has historically recovered around 30% of the total gold ounces recovered. This Feasibility Study considers the inclusion of a cleaner cell to increase the LOM average recovery through flotation to 31.9%. After expansion, the mill is expected to have an overall recovery of 94.4%.

In addition to the operational performance that underpins the basis of design for the mill expansion from 800 t/d to 1,200 t/d, there has been extensive additional testwork completed in 2020 to further evaluate opportunities and refine the scope of the expansion. Recent testwork included:

- Comminution testwork on Sugar and Middle Zone ore—performed by Met-Solve Laboratories Inc. (Met-Solve)
- Cyanide leaching testwork on flotation and gravity concentrate—performed by SGS Lakeside
- Tailings thickening testwork—performed by Outotec (Canada) Ltd.

The Feasibility Study evaluated multiple flowsheet options for the Expansion:

- Gravity concentration followed by rougher and cleaner flotation, and sale of doré bars and concentrate (closest to the existing circuit)
- Gravity concentration followed by rougher flotation and leaching with Merrill-Crowe (MC) gold recovery of the re-ground flotation concentrate and sale of solely doré bars
- Gravity concentration followed by rougher flotation and leaching with carbon-in-leach (CIL) gold recovery of the re-ground flotation concentrate and sale of solely doré bars

On the basis of recent and previous metallurgical testwork and an economic trade-off, Harte Gold has decided to retain the existing mill flowsheet and expand the circuit's capacity; the only notable addition to the existing flowsheet will be cleaner flotation, which will improve circuit operability, increase overall recovery, and improve the grade of the flotation concentrate that will be sold.

1.11 Mineral Resource Estimate

The 30 September 2020 Mineral Resource estimate was carried out by the Harte Gold Vice President of Geological Services, Mr. Vincent Cardin-Tremblay, P.Geo. Mr. Cardin-Tremblay is considered a QP within the meaning of the Canadian Securities Administrator's National Instrument 43-101.

This Mineral Resource evaluation represented a change in methodology from the previous resource estimate. The previous Mineral Resource estimate, dated February 2019, used 1 m composites and a 1.5 m mining width to perform a multiple indicator kriging estimation of the domains. The current Mineral Resource estimate uses full-width composites (FWC) and no minimum mining width to perform a 2-D kriging estimation (or 2-D inverse distance cubed for some domains where not enough composites were available to justify using kriging). The 2-D kriging with FWC approach offers numerous advantages with narrow orebodies:

- Simplifies the compositing decisions as there is only one composite per intersection
- Removes one dimension from the problem, which simplifies the geometry and calculations
- Avoids the challenges that come from defining 3-D search orientation and anisotropy for very-thin domains
- Does not assume selectivity within the width of the domains, as their average width is often narrower than the true minimum mining width (1.8 m).

The application of 2-D kriging with FWC was carried out between April and June 2020. It was a multi-step process that started with checking and validating the geological database, as well as reviewing the domain interpretations and creating the vein models (true mineralized thickness). The vein models were then used to code and generate the FWC, and capping was applied on the FWC rather than on individual assays.

One consequence of estimating with 2-D kriging is that grade cannot be estimated directly; rather, it must be calculated based on the "accumulation" and thickness to comply with the geostatistical requirements of additivity and constant support. Accumulation in this context refers to the grade multiplied by the thickness of the FWC, and it is calculated for each composite in a three-step process:

- True thickness of each point extracted from the modelled wireframes of each domain
- True thickness value attributed to each FWC
- Accumulation calculation = grade of FWC x true thickness of FWC.

The accumulation and thickness attributes of each FWC become the estimation variables, and variography of both variables is used to create the rules for creating the rotated block model. In this Mineral Resource estimate, dynamic anisotropy (DA) was used in the estimation to address the local changes in orientation of the modelled vein wireframes. Once the block model and estimation were created, the accumulation estimate for each block was divided by its thickness estimate to derive the block's grade.

Based on previous estimates and testwork, a global bulk density of 2.73 t/m^3 was used in the Mineral Resource estimate.

The cut-off grade for the Mineral Resource estimate was applied as a grade-thickness cut-off grade to ensure that not only did blocks meet the grade requirements, but also that when extracted at a minimum stoping width of 1.8 m the estimate would represent the material with a reasonable potential for economic extraction. The economic parameters for the cut-off grade calculation are detailed in Table 1-1.

Table 1-1: Economic Parameters for Mineral Resource Estimate Cut-Off Grade

Parameter	Value
Gold Price	US$1,600/oz Au
US$:C$	1.00:0.75
Net Smelter Royalty	3.5%
Treatment/Recovery Cost	$50/oz Au
Process Recovery	94.0%
Payable Gold	98.0%
Mining, processing, G&A Cost	$203/t

The resulting cut-off grade is 3.0 g/t Au; when multiplied by the minimum mining width of 1.8 m it results in a grade-thickness cut-off of 5.4 m*g/t Au.

The Mineral Resource was depleted to 30 September 2020 and is given in Table 1-2.

Table 1-2: Mineral Resource Estimate at 3.0 g/t Au Cut-off Grade and 5.4 m*g/t Grade-Thickness Cut-off

Classification	Zone	Tonnes ('000s)	Au (g/t)	Au (oz '000s)
Indicated	Sugar (North and South)	1,315	15.53	657
	Middle	1,326	8.95	381
	Wolf	162	6.13	32
	Total	**2,803**	**11.87**	**1,070**
Inferred	Sugar (North and South)	891	10.98	314
	Middle	844	8.21	223
	Wolf	132	7.04	29
	Total	**1,866**	**9.45**	**567**

Notes: The stated Mineral Resources comply with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and are classified in accordance with the *CIM Definition Standards for Mineral Resources & Mineral Reserves* (CIM Definition Standards).
The Mineral Resource is inclusive of the Mineral Reserves.
The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
The Mineral Resource estimate was prepared by Mr. Vincent Cardin-Tremblay (P.Geo.), former VP Geological Services for Harte Gold. Mr. Cardin-Tremblay is a QP as defined by NI 43-101.
Mineral Resources have been estimated as of 30 September 2020 using a gold price assumption of US$1,600 per ounce Au. A resource cut-off grade of 3.0 g/t Au has been estimated based on operating cost projections and applicable metallurgical recovery. The cut-off grade was used in combination with a minimum mining width factor of 1.8 m to define the resource.
Totals may not add due to rounding.

Validation of the Mineral Resource estimate was done with several industry-standard methods, including:

- Visual validation comparing block estimates to sampling on sections and plans.
- Global comparison to de-clustered distributions. This involves comparing mean gold grades between estimated blocks and the nearest neighbour (NN)estimate.
- Comparison to previous resource estimates.
- Comparison to ordinary kriging, (OK), inverse distance squared (ID2), and NN estimations.
- Global swath plots comparing estimates to composites on a more local scale, with elevation, northing, and easting.
- Evaluation of the effect of grade capping.

The QP considers the Mineral Resource estimate to be reasonable and in conformity with the generally accepted *CIM Estimation of Mineral Resources & Mineral Reserves Best Practices Guidelines* (CIM, 2019).

1.12 Mineral Reserve Estimate

The Mineral Reserve estimate for the Sugar Zone Mine, effective 31 December 2020, comprises an estimated 3.45 Mt of ore at a gold grade of 7.18 g/t Au, and contains 797,000 oz Au, as shown in Table 1-3.

Table 1-3: Mineral Reserve Estimate as of 31 December 2020

Classification	Zone	Tonnes ('000s)	Grade (g/t)	Ounces ('000s)
Probable	Sugar (North and South)	1,994	7.59	487
	Middle	1,460	6.62	311
	Total	**3,454**	**7.18**	**797**

Notes: The stated Mineral Reserves comply with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and are classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's CIM Definition Standards for Mineral Resources & Mineral Reserves. Mineral Reserve estimates reflect the Company's reasonable expectation that all necessary permits and approvals will be obtained and maintained.
Mineral Reserves are the economic portion of the Indicated Mineral Resources. Mineral Reserve estimates include mining dilution at grades assumed to be zero.
The 2020 Mineral Reserve estimate was prepared under the supervision of Mr. Chris McCann (P.Eng.), Director of Technical Services for the Company. Mr. McCann is a QP as defined by NI 43-101.
The Mineral Reserves were estimated as of 31 December 2020 using a gold price assumption of $1,450/oz.
A mining cut-off grade of 5 g/t has been estimated based on operating cost projections, sustaining capital development cost, mining dilution and recovery, royalty payment requirements, and applicable metallurgical recovery.
Totals may not add due to rounding.

The Harte Gold mine engineering and geology team worked with Deswik Consulting to generate the Mineral Reserve estimate, and the methodology and execution were supervised by, and validated by Mr. Chris McCann, P. Eng., Director of Technical Services for Harte Gold. Mr. McCann is considered a QP within the meaning of Canadian Securities Administrator's National Instrument 43-101.

To create the Mineral Reserves, the mineral resource block model was processed through the Deswik Stope Optimizer (DSO) to create potentially economical stoping blocks above the marginal cut-off grade of 3.5 g/t Au, which were generated at a minimum horizontal width of 2 m and a strike length of 10 m to ensure close conformity to the mineralization. The DSO stope shapes were then smoothed to improve their geometry for extraction; this process included aggregating the created shapes to stope lengths of up to 60 m in the Sugar Zone and up to 30 m in the Middle Zone. An additional process of dilution and recovery was then applied to the stope shapes created, ensuring that all stopes had a minimum diluted width of 2 m of true thickness; stope recovery was estimated to be 95% for non-sill pillar stopes; and sill pillar stopes were estimated at 85% recovery. All sill drift development assumes 100% recovery, and the sill drift dimensions consider an average of 10% dilution above the minimum required drift dimensions. All material outside of the block model is considered as having no grade.

The cut-off grade to create the stope shapes was assessed to be 3.5 g/t Au. More-detailed cost analysis demonstrated that sill development had an incremental cut-off grade of 1.5 g/t Au; the overall cut-off grade for each mining block as a whole was approximately 5.0 g/t Au when considering the expected 1,200 t/d cost profile as determined in this Feasibility Study. This overall break-even cut-off grade is inclusive of pre-production capital development. Table 1-4 provides a summary of the cut-off grade calculations used for the Mineral Reserve estimate. During the extraction sequencing process, the break-even cut-off grade of 5.0 g/t was used to evaluate the mining blocks and the overall mine plan by period.

Table 1-4: Break-Even and Incremental Cut-Off Grade Estimates After Expansion (2023–2027)

			Unit	Break-Even Cut-off Grade	Marginal Cut-off Grade	Incremental Dev. Cut-off Grade
Unit Costs	Operating Cost	Mining	$/t	130	110	0
		Processing	$/t	34	34	34
		G&A	$/t	44	44	44
	Sustaining Capital	UG dev.	$/t	59		
	Total		$/t	**267**	**188**	**78**
Gold Price			US$/oz	1,450	1,450	1,450
Exchange Rate			US$:C$	0.76	0.76	0.76
Process Recovery		94.4%	$/oz	(107)	(107)	(107)
Payable Gold		98.3%	$/oz	(32)	(32)	(32)
Treatment/Recovery Cost			$/oz	(50)	(50)	(50)
Royalty			$/oz	(67)	(67)	(67)
Net Revenue			$/oz	1,652	1,652	1,652
Net Revenue/Gram Au			$/g	53	53	53
Cut-off Grade			**g/t**	**5.0**	**3.5**	**1.5**

1.13 Mining Methods

The Sugar Zone Mine mineralization typically comprises tabular, narrow veins, with economic resource widths varying from 0.2 to 4.4 m and dipping at around 70° to the southwest. Ground conditions are exceptionally good, allowing narrow longhole stopes to be mined with relatively low levels of unplanned dilution.

Access to the mine is via a surface portal and a single 4.3 m wide x 4.5 m high decline to about 100 m below the portal, where the main decline divides into three separate footwall (FW) declines that provide access to the North, South, and Middle Zones.

A standard fleet of mechanized equipment is employed in development and longhole open stoping, comprising two-boom and single-boom drill jumbos, 1.8 m^3 (2 yd) and 4.5 m^3 (6 yd) load-haul-dump (LHD) loaders, 30-tonne trucks, scissor decks, longhole drills, and various ancillary equipment.

In late 2020, the operation transitioned from contractor to Owner mining for development and mining operations. Longhole drilling and blasting and definition drilling continue as contractor operations.

The mine is forecast to produce at a rate of 800 t/d of ore in 2021 and to increase to 1,200 t/d by Q1 2023; the ramp-up in production is currently scheduled to occur during Q4 2022. This increase in underground production will be driven primarily by full production rates being reached in the Middle Zone, which is being accessed and developed in late 2021 and throughout 2022. The increase in underground production is also timed to coincide with the completion of the process plant expansion, as there is limited space to build up a large surface stockpile. Paste backfill is planned for implementation in early 2023 and will reduce the need for additional mobile equipment currently required to place unconsolidated waste rock backfill into mined-out longhole stopes.

The LOM production plan (Figure 1-2 & Figure 1-3) shows that ore production is sustained through 2027, after which time the known Mineral Reserves begin to become a constraint for increased throughput, and the tonnage and grade ramps down in 2028 and 2029 as remnant stopes close to surface, and deeper, sequence-constrained Mineral Reserves are then extracted. Extending the mine life through mine exploration activities adding Mineral Reserves is considered probable, however this has not been factored into this LOM production plan for expansion to 1,200 t/d.

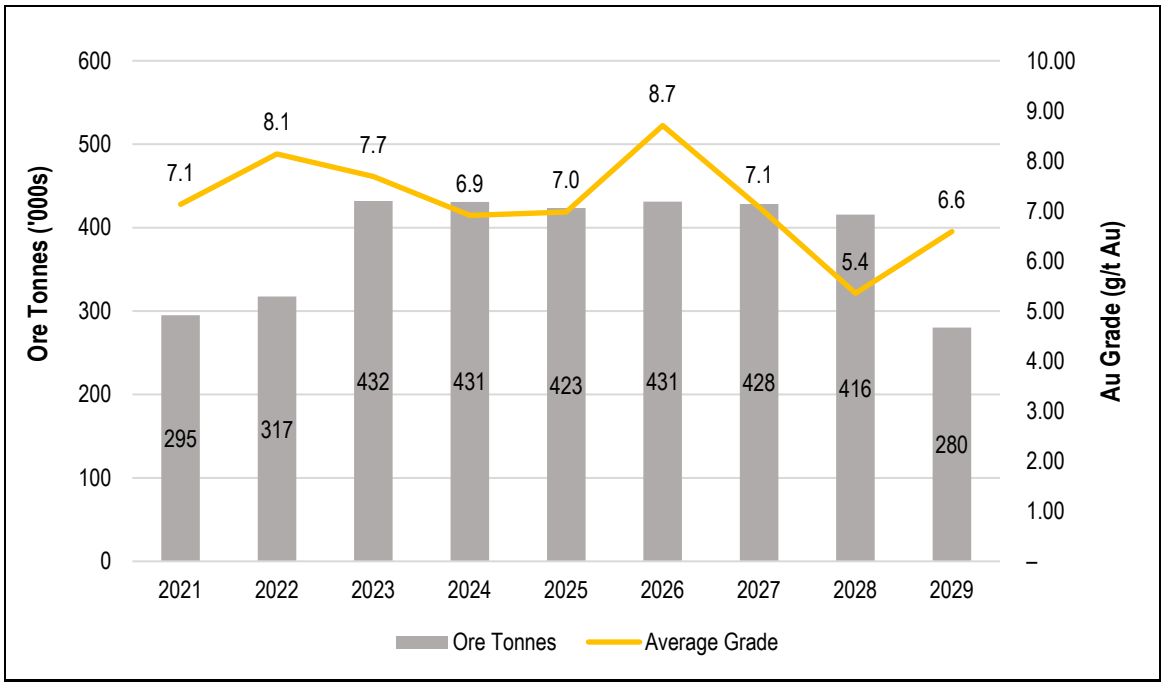


Figure 1-2: Mined Ore Production and Gold Grade Profile by Year

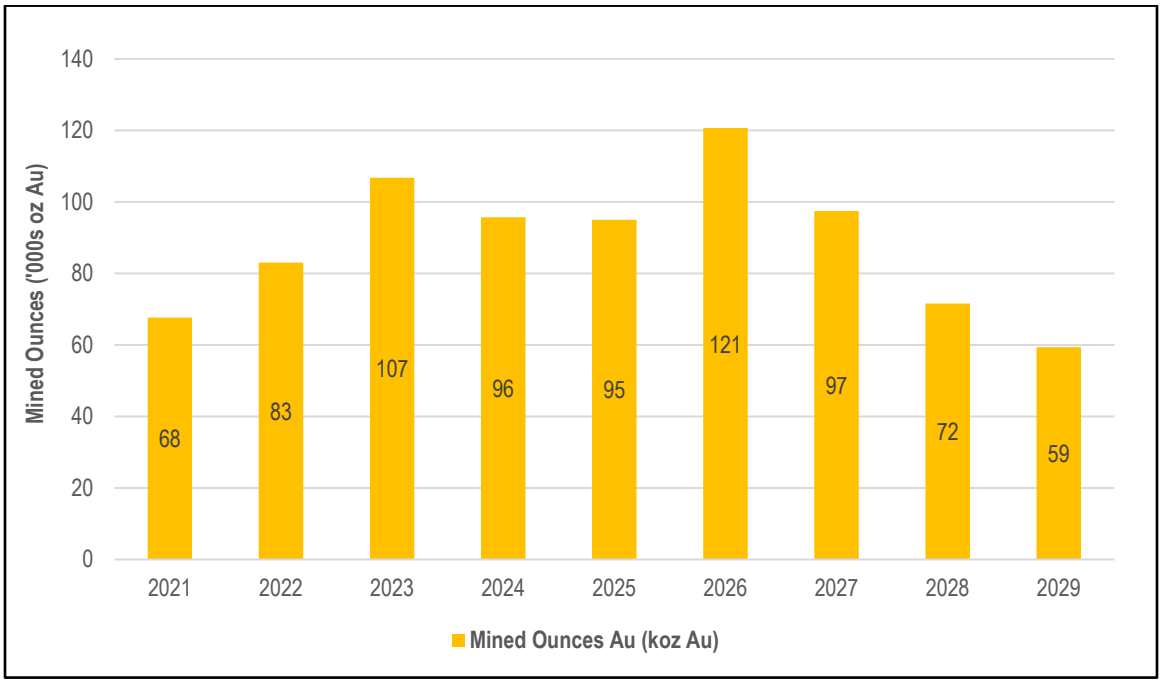


Figure 1-3: Mined Contained Gold Ounces by Year

A critical component of the mine production plan is a continued focus on pre-production capital development. This development consists of the vertical extension of each of the three ramps, as well as sub-level accesses, remucks, truck load-outs, ventilation drifts and raises, and all other development required to support the safe production of longhole tonnes. The LOM plan includes approximately 34,000 m of capital development which peaks in 2021 at nearly 5,000 m, then remains relatively constant through the remainder of the LOM at approximately 4,000 m/a.

Additional underground development is required to create the over- and under-cuts of the longhole stopes, termed operating sill development (“sill drifts” or “operating development”). The LOM plan includes approximately 36,000 m of operating development across the three major mining zones. Annual operating development quantities average 4,000 m/a over the nine-year mine life but average closer to 4,700 m/a from 2022 through 2025 as the mine expands production by increasing the number of available stoping horizons. Figure 1-4 provides a summary of capital and operating development over the LOM.

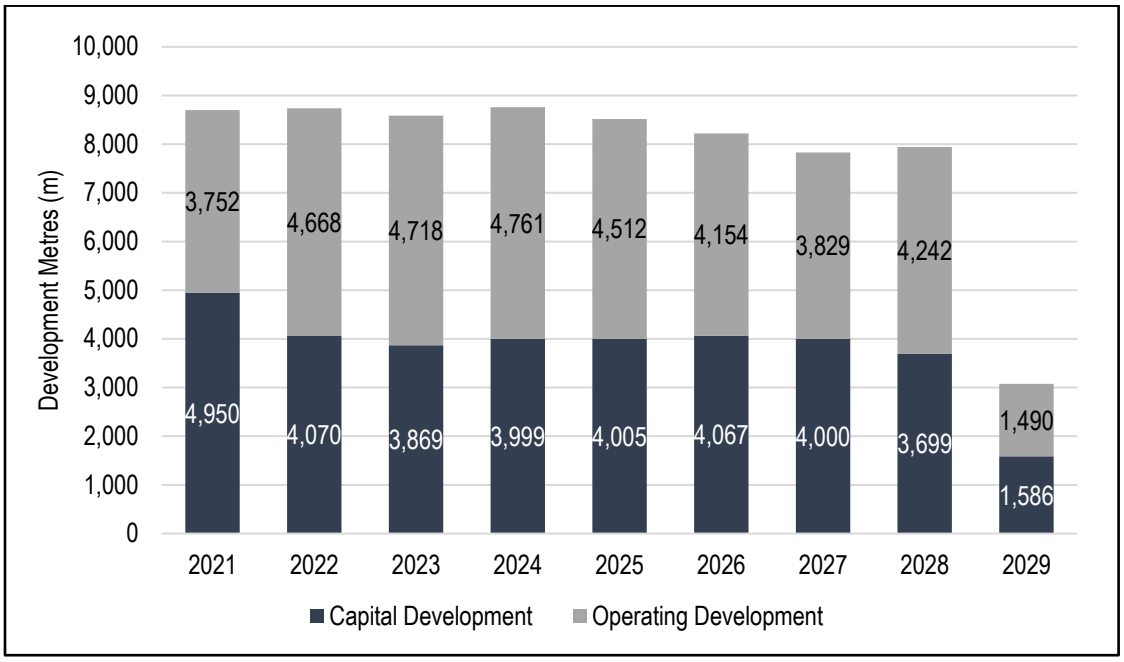


Figure 1-4: Summary of Lateral Capital and Operating Development by Year

To achieve a production rate of 431,000 t/a (1,200 t/d), a relatively consistent total material movement (ore + waste) of approximately 670,000 t/a is required for the period from 2023 through 2028 (Figure 1-5). Figure 1-6 provides a visual representation of the LOM extraction and development sequence.

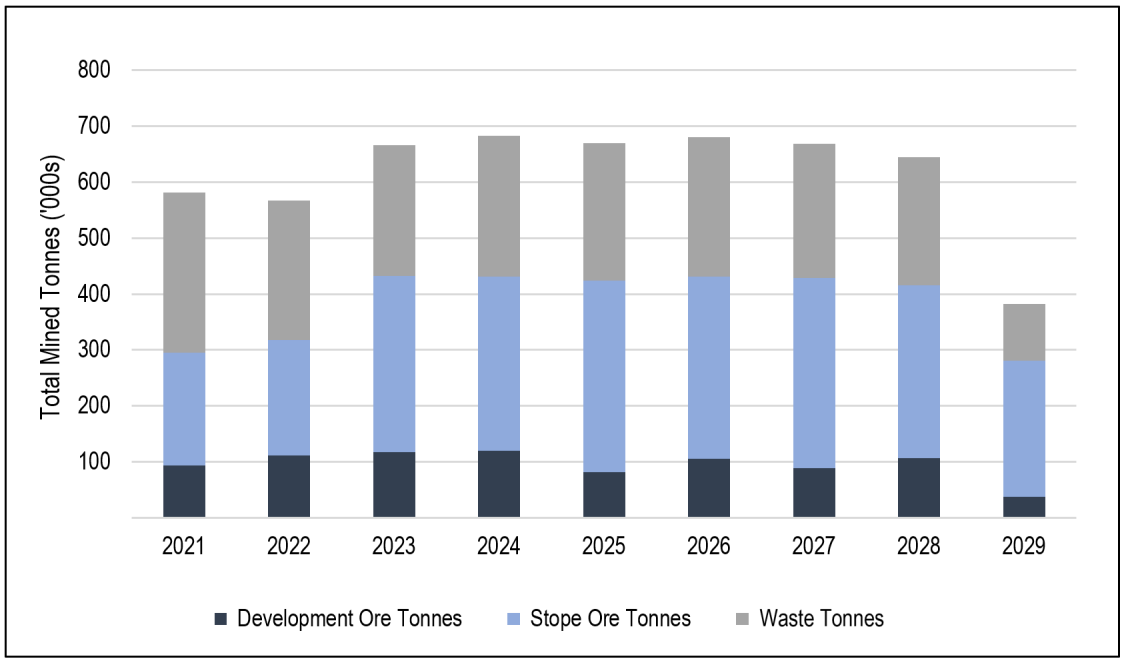
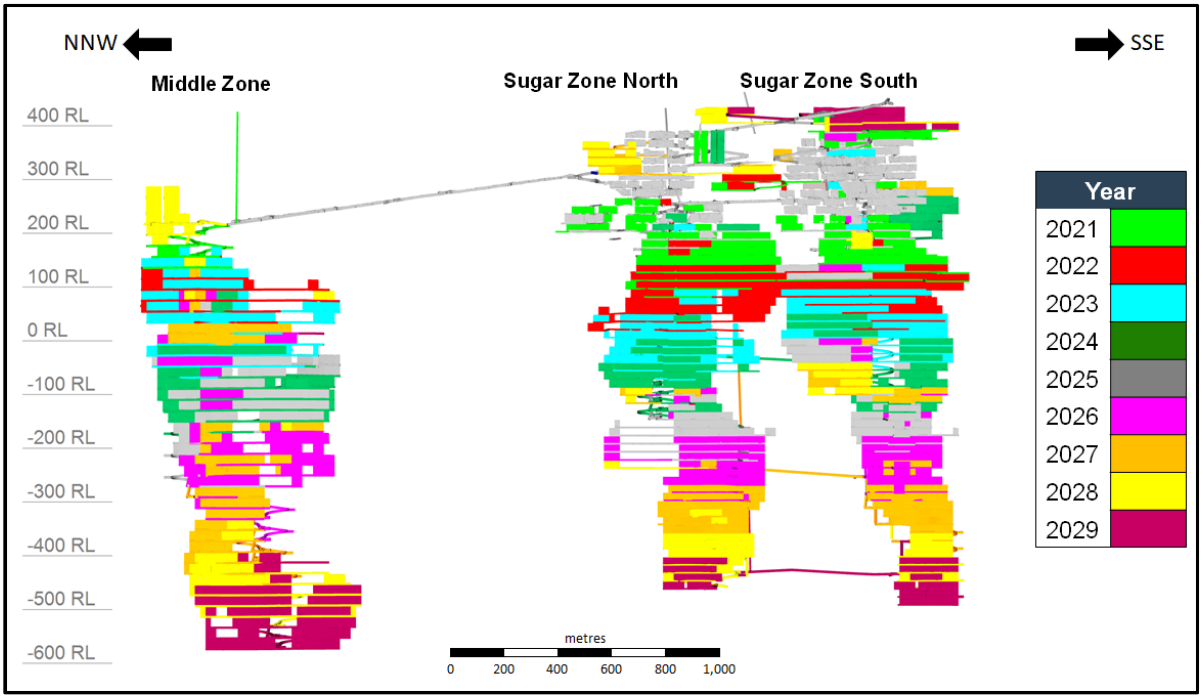


Figure 1-5: Total Material Mined by Year



Note: The vertical scale of this figure is in metres relative to sea level elevation (RL).

Figure 1-6: LOM Extraction and Development Schedule

During 2021 and 2022, stopes will be backfilled with unconsolidated waste rock generated by the capital development; this method has been used successfully since the start of stoping operations. At the beginning of 2023, paste backfill is planned to become the primary method of backfilling open stopes.

The switch to paste backfill will require modifications to the paste backfill plant, located within the mill footprint, as well as the construction of an underground paste distribution system (UDS). In the paste backfill plant, the major equipment change will be a replacement of the current ceramic disc filter with a higher-capacity cloth disc filter. This change in filter technology and capacity will allow for the paste backfill plant to produce continuous full-feed (1,200 t/d) filter cake for the production of paste backfill underground, and for filter cake for transport (ie. dry stack) to the North Tailings Management Facility (North TMF) until commissioning of the South Tailings Management Facility (South TMF) in Q2 2025. In addition to the change in filter technology, the paste backfill plant will also house a hydraulic-piston paste pump and a high-pressure centrifugal flush pump, which will form the start of the UDS.

The UDS will connect the paste backfill plant to each individual underground stope using the paste pump. The pump will feed a 10 cm DN100 pipeline to a lined borehole that will be drilled from surface, with outlets on two separate underground horizons allowing for efficient paste-pipe reticulation to the North, South, and Middle Zones. Due to the lateral distance between the Sugar and Middle Zones, an underground paste booster-station will be required, and this will include an additional hydraulic-piston paste pump and a high centrifugal flush pump. The full initial construction of the UDS is

expected to occur between 2022 and the first half of 2023, with an initial focus on the North and South Zone distribution system followed by completion of the Mid-Zone booster station and distribution piping. Throughout the LOM, the UDS will be advanced vertically down the three zones, and temporary sub-level piping will be installed in the sill drifts to reach each stope in the mine plan.

1.14 Process Plant

The existing Harte Gold Sugar Zone process plant currently produces gold doré bullion and gold flotation concentrate through gravity concentration and flotation circuits, respectively. The process plant was commissioned in late-2018 and was designed to produce at 800 t/d. Metallurgical testwork and trade-off studies recommend that the current mill flowsheet be broadly maintained, but at higher throughput, with the notable addition of a cleaner flotation cell. The expanded process plant will continue to produce gold doré bars and gold flotation concentrate.

The Feasibility Study used the current performance conditions and key performance indicators (KPIs) to calibrate predictive comminution circuit models. These models identified that the existing secondary cone crusher and the existing 800 kW ball mill are the primary bottlenecks at increased throughputs. To increase mill throughput by 50%, several key components and circuits will need to be expanded:

- Replace HP100 cone crusher with HP200 (or similar) cone crusher
- Add a 550-kW ball mill in series with the existing primary ball mill
- Add a cyclone bank to ensure that there are always two cyclones in operation
- Add two rougher flotation cells
- Add one new cleaner flotation cell
- Add tailings thickener
- Expand gold room.

Besides the major components listed above, the upgrade in mill throughput will require changes to pumps, pump boxes, piping, and other support systems.

The new cone crusher and additional ball mill will require extending the crusher and mill buildings, and the Project schedule has the foundation preparation work beginning in Q2 2022, as soon as the ground has thawed, and conditions permit. There is opportunity to complete some or all of the building foundation work in the 2021 construction season (before October) and de-risk the remainder of the process plant expansion schedule. The process plant will continue operating during construction, with limited well-planned shut downs for tie-ins of new equipment.

As the process plant will maintain its current flowsheet, and comminution and recovery technology, Harte Gold does not anticipate any material impact to process plant throughput or recovery prior to or after the expansion to 1,200 t/d.

1.15 Site Infrastructure

1.15.1 Site Surface Infrastructure

The site is accessible via the all-weather road network of the White River Forest Sustainable Forest License area that connects to Highway 631 and Highway 17. On-site administration offices are fully operational. A microwave tower provides a communication link to White River and local cell-phone coverage.

Electric power is provided by a 45 km long, 25 kV wooden pole line that extends along road corridors from the mine site to the White River water treatment area beside Highway 631, north of the town of White River. Electricity is distributed from a central substation at the mine site with step-down transformers. A back-up generator is used to supply electricity for critical process plant operations in an interruption to electrical service.

The throughput expansion will require an extending the existing process plant complex including an additional 700 m^2 of floor space shared amongst the crusher and mill buildings, gold room, and electrical room. Additional process plant equipment outside of current building footprints includes a 9 m diameter tailings thickener and a process-water tank.

Additional surface infrastructure required over the LOM includes a maintenance workshop, a second fresh-air raise with fan and heater, and a containerized extension to the effluent treatment plant (ETP).

Ancillary buildings are generally portable and/or modular structures, and consist of a gatehouse, coreshack, core storage, assay lab, maintenance shop, warehouse, ventilation fan buildings, modular office and dry, and electrical infrastructure. Explosives are delivered to underground storage areas on an as-needed basis, and no storage magazines on surface are required.

1.15.2 Tailings and Water Management

Tailings deposition at the Sugar Zone Mine will occur in three forms:

1. Hydraulic slurry tailings—42%
2. Filtered tailings for stacking inside the North TMF footprint—33%
3. Paste backfill to fill underground voids—25%.

Currently the Operation is using the existing North TMF, which has a crest elevation of 406.0 masl, which will be raised to a final crest elevation of 412.6 masl after interim lifts in 2021 and 2023. The North TMF will be used for a combination of slurry and filtered tailings deposition and will be used until Q2 2025, after which all tailings will be deposited in the South TMF as slurry. The North TMF is anticipated to store 1.3 Mt of dry tailings material. All permits for future lifts of the North TMF are in place and considered in the current Closure Plan (Harte Gold Corp., 2020).

The South TMF will store a total of 1.2 Mt of dry tailings; the construction of the initial lift will occur in 2024. The South TMF will take advantage of a natural basin, with the topography acting as the impoundment structure on three sides, and a single dam located on the northern border of the facility. The initial lift of the South TMF dam will see a crest height at 406.4 masl and will reach a final crest height of 411.0 masl after interim and final lifts in 2026 and 2028. The South TMF location has received preliminary location approval from the Ministry of Energy, Northern Development and Mines (MENDM), but the facility will need to receive "Plans and Specifications Approval for Stage 1 South TMF" prior to its inclusion in a future Closure Plan Amendment.

Current and future dam structures have been and will be designed and constructed in accordance with Canadian national (Canadian Dam Association) and provincial (*Lakes and Rivers Improvement Act*) guidelines. Consideration has also been given to the *Global Industry Standard on Tailings Management* (GISTM) for some design elements. GISTM requires that all dams have closure designs that can withstand the 1-in-10,000-year design earthquake, and all TMF structures at the Sugar Zone Mine are designed to this standard to minimize the structural modifications required at closure.

Tailings from the process plant are thickened prior to deposition in the TMF, allowing improved recovery of process water (including reagents and heat) within the process plant building and also increasing the dry settled density of tailings solids in the TMF. This practice will reduce the size of the reclaim pond, reduce liquid-solid separation following deposition in the TMF, improve the physical stability of the TMF, and reduce both the risk and consequences associated with a potential failure of a perimeter embankment.

Tailings-delivery pipelines and reclaim-water pipelines for the North TMF are located within the mine site watershed or the TMF watershed. Any potential spillage from the pipelines would drain by gravity to the TMF or to the Operation site.

Tailings-delivery pipelines and reclaim-water pipelines for the South TMF will follow the access road corridor and will be designed and constructed with industry-standard spill containment systems.

Stormwater reports to either the mine-site containment pond or to the reclaim pond of the TMF. Surplus water from the containment pond is pumped to the TMF and/or to the overland flow-path north of the underground portal location that drains to the TMF to attenuate some ammonia (from mine explosives) found in mine water. No untreated storm water discharge from the Operation is anticipated.

Groundwater is drawn from a well at the mine site and treated in a containerized potable water treatment plant prior to use.

Domestic sewage is collected and treated in a containerized Waterloo Biofilter sewage disposal system located at the mine site. Treated water from the Waterloo Biofilter is pumped to the TMF for inclusion in water recycled to the process plant.

Currently, the Sugar Zone Mine has a maximum water discharge allowance of 2,100 m^3/d and water is discharged to Gagegenha Lake via an underwater diffusor. The discharge season at the Sugar Zone Mine begins when only 10% of the surface of Gagegenha Lake is under ice cover, and ends at the end of November. All water is treated through the existing ETP, which is currently capable of treating up to 1,500 m^3/d to the required effluent quality. Over the next 18 months Harte Gold will implement upgrades to the ETP to improve its capacity to remove the ammonia mass load from the underground mine water, and to remove suspended solids such as algae which grows in the reclaim pond during summer months. The planned capacity improvements in the ETP will be supplemented by mechanical evaporators located at the North TMF; this will ensure that the site is able to maintain a neutral water balance with no year-over-year accumulation of water inventory.

Upon commissioning of the South TMF, the required discharge rate will increase from 2,100 m^3/d to 4,100 m^3/d due to the large catchment area of the South TMF. To facilitate this increased discharge rate, an additional containerized module of the ETP will be required. This will treat the water from the South TMF in a two-stage process involving precipitation of metals via reagent injection and reaction tanks, and metal precipitates and suspended solids will be removed via multi-media filtration.

1.16 Market Studies and Contracts

No market studies were conducted by Harte Gold. Gold is a freely traded commodity on the world market for which there is a steady demand from numerous buyers.

Harte Gold presently produces gold in both gold doré bullion and gold flotation concentrates.

Gold doré bars are presently sent to Asahi Refining Canada Ltd. (Asahi), in Brampton, Ontario, for refining. Transportation and refining charges have been agreed to and are typical of charges in the industry.

In respect of refined gold produced from doré bars, Harte Gold has three off-take agreements pursuant to which Harte Gold is obliged to deliver a portion of the resultant ounces until the total number of ounces required to be delivered have been delivered. In particular:

- Orion Mine Finance (Orion) is entitled to 50% of the refined doré ounces until 375,000 oz have been delivered.
- ANR Investments B.V. (Appian) is entitled to (i) 11.5% of the refined doré ounces until 93,750 oz have been received, and (ii) 18.5% until 500,000 of the refined doré ounces have been delivered.

Each of Orion and Appian then pay Harte Gold for such ounces based on the gold price quoted by the London Bullion Metals Association (LBMA) .

Harte Gold has entered into a contract with Glencore Canada Corporation (Glencore) to sell gold concentrates to Glencore's Horne Smelter in Rouyn, Québec. The term of the contract runs until 2030,

cancellable any time after February 2020 with six months' notice. Glencore purchases the gold concentrates on arrival at the Horne Smelter at the market gold price less treatment and refining charges. The gold price is based on the gold price quoted by the LBMA.

Harte Gold has entered into an Impact Benefits Agreement (IBA) with the Pic Mobert First Nation (PMFN), the proximal First Nation, in connection with the Company's Sugar Zone Property. The IBA applies to all mines that may be developed on the 81,287 ha Sugar Zone Property, and provides the framework within which Harte Gold and PMFN will continue to work together during the production phase of the Sugar Zone Mine. The key IBA financial terms include a 4% net profits interest (NPI), based on the World Gold Council definition of all-in sustaining cost metrics.

1.17 Environmental Studies, Permits, and Social, or Community Impact

Harte Gold is committed to respecting the natural environment and maintaining strong community partnerships. We are committed to operating our facility within federal and provincial regulations, and fulfilling our commitments to our First Nations partners. From exploration to mine closure, Harte Gold strives to employ best-management practices to prevent or mitigate any potential environmental impacts.

The Sugar Zone Property is in an area of northern Ontario that has hosted mineral exploration and mining projects for more than 50 years. The Property has a history of gold exploration since 1969. Previous exploration activities on the Property have comprised surface drilling, geophysical surveys, geological mapping, and exploration development. No environmental liabilities exist related to previous exploration activities.

To realize the throughput expansion, Harte Gold must amend several existing permits and obtain new permits for the South TMF. The permitting timeline for the expansion is between 12 and 18 months and will involve community and First Nations consultations. Table 1-5 provides a summary of the permitting roadmap and the associated regulatory agencies and legislation.

The proposed expansion to 1,200 t/d will not trigger a federal impact assessment as per Section 19 of the Physical Activities Regulations.

Harte Gold has completed provincial class environmental assessments (EA) for Resource Stewardship and Facility Development, in accordance with MNR (2003). The Project is not required to complete any additional provincial EA processes.

Table 1-5: Permits Required for Expansion

Permit / Permit Amendment	Regulatory Agency	Relevant Legislation	Rationale for Permit Issuance	Rationale for Amendment
Amendment to Sewage Environmental Compliance Approval 8479-B3QHJZ	Ministry of Environment, Conservation and Parks	*Environmental Protection Act*	Approves industrial and domestic sewage works	Upgrade to effluent treatment plant and increased effluent discharge volume, development of South TMF
Amendment to Air Environmental Compliance Approval 7924-BBLL6K		*Environmental Protection Act*	Approve site wide air emissions	Increased mine and mill production rates and associated modifications to air emission sources
Amendment to Sugar Zone Project Closure Plan	Ministry of Energy, Northern Development and Mines	*Mining Act*	Approve development and closure of the production of the Project	Increased production rate; changes to site footprint including development of the South TMF and additional financial assurance
Plans and Specifications Approval for Stage 1 South TMF	Ministry of Natural Resources and Forestry	*Lakes and Rivers Improvement Act*	Approves development of South TMF	Approves construction of the Stage 1 South TMF; approvals for subsequent stages to be obtained on an as-needed basis
Forest Resource License		*Crown Forest Sustainability Act*	Permit not yet issued	Approve the harvest of Crown owned merchantable timber in the area of the South TMF

Harte Gold has necessary permits for an 850 t/d mine and a 950 t/d process plant. Permit amendment applications are currently being prepared for submission starting in Q1 2021 related to a production rate increase to an average of 1,200 t/d. The Ministry of Natural Resources and Forestry and the MENDM have indicated that no further *Lakes and Rivers Improvement Act* approval is required for the existing North TMF. The Company is advancing baseline monitoring at the proposed South TMF location, and the design will be advanced to the issued for construction design stage for the Closure Plan amendment.

Harte Gold has completed a closure plan incorporating baseline environmental monitoring. The Company has also developed an environmental management system for the Operation.

Amec Foster Wheeler (2016d) characterizes the Operation as generally having a low risk of acid generation and metal leaching.

Water discharge is minimized by recycling water to the extent practical. Surplus water from the TMF that is not needed for the process plant is treated and discharged to an underwater diffuser in Gagegenha Lake that has been engineered for efficient mixing. All water discharged from site is below the contaminant thresholds as defined in the Sugar Zone Sewage Environmental Compliance Approval.

Harte Gold has undertaken consultation with First Nations on behalf of the Crown, and as directed by MENDM regarding exploration and development activities on the Property since June 2010. Consultation to date with First Nations has not identified the presence of cultural heritage values that would be affected by the Operation.

The Crown delegated procedural aspects of consultation to Harte Gold for PMFM and the Biigtigong Nishnaabeg (Ojibways of Pic River Heron Bay) First Nation (PRFN). Harte Gold has entered into an IBA with PMFM and a Memorandum of Understanding with PRFN to establish strong working relationships.

1.18 Capital and Operating Costs

1.18.1 Capital Costs

This Technical Report covers an expansion scenario at the Sugar Zone Mine, and capital expenditures have been divided into two distinct categories: expansion capital and sustaining capital.

Expansion capital expenses are defined as those which allow the Operation to increase throughput from the current 800 t/d to 1,200 t/d. As the Project does not consider the inclusion of any new mineralized zones or change the mineral reserve base of the deposit from the current 800 t/d operation, the expansion capital is limited to process plant expansion, mining mobile fleet expansion, and permitting. Expansion capital expenditures are forecast to begin in Q3 2021 and end in Q1 2023, once the process plant has been commissioned and the mining fleet is on site and producing 1,200 t/d sustainably.

Sustaining capital expenditures relate to sustaining the existing production and allowing Harte Gold to mine its current Mineral Reserves during and post the Project period. It is important to note that due to the expansion in throughput, some of the sustaining capital expenditures are accelerated, and are incurred earlier in the mine life than they would have been if the mine had continued at the current 800 t/d throughput, but the scale and nature of the expenditure does not change materially due to increased throughput. Table 1-6 summarizes the total capital costs.

Table 1-6: LOM Capital Expenditure

	Total ($ M)
Expansion Capital	21
Sustaining Capital	325
Total Capital	**346**

The expansion capital of $21.1 million will be incurred from Q3 2021 to Q1 2023. Expansion capital comprises process plant expansion ($16.3 million), mine equipment ($4.6 million), and permitting ($0.1 million). The expansion capital expenditures are summarized by period in Table 1-7.

Mill expansion capital includes but is not limited to:

- Replacing existing secondary cone crusher with a larger unit
- Adding a 550-kW ball mill in series with existing primary ball mill

- Adding a cyclone bank to ensure that there are always two cyclones in operation
- Adding two rougher flotation cells
- Adding one new cleaner flotation cell
- Adding tailings thickener
- Expanding gold room.

Mine equipment includes purchasing additional equipment to accommodate increased tonnage and working faces. Key equipment includes:

- One additional 30-tonne haul truck
- Three additional LHD loaders
- Ancillary equipment.

Permitting capital includes consultant costs for amendments to existing permits and other miscellaneous costs. Expansion capital expenditures start in Q3 2021, with the bulk of the expenditures occurring in 2022.

Table 1-7: Expansion Capital by Year

	2021 ($ M)	2022 ($ M)	2023 ($ M)	LOM ($ M)
Mill Expansion	2.4	14.0	0.0	16.3
Mine Equipment	0.0	2.3	2.3	4.6
Permitting	0.1	0.0	0.0	0.1
Total	**2.5**	**16.3**	**2.3**	**21.1**

Note: Totals may not add due to rounding.

Sustaining capital expenditures total $325 million between 2021 and 2029, as summarized in Table 1-8. Underground development, makes up the largest portion of the LOM sustaining capital, with over 34,000 m of lateral development planned at a LOM average of approximately $5,700/m. Sustaining capital expenditures also include:

- Underground infrastructure capital—paste distribution (sustaining), mine ventilation, underground shops and infrastructure
- Tailings and water management—expansion of the North TMF, construction of the South TMF, and water management
- Mine equipment maintenance—overhaul and major component replacement of the equipment fleet
- Underground electrical—distribution of electrical equipment for ventilation, drilling, and dewatering

- Paste backfill—substitution of the existing ceramic disk filtration system with a cloth disk filter providing 1,200 t/d capacity, underground distribution system with positive displacement pump on surface, and installation of paste booster station for the Middle Zone
- Reclamation / Closure Costs: Incurred at the end of mine life
- Other—project management and indirect costs.

Table 1-8: LOM Sustaining Capital

	LOM Total ($ M)
Underground Development	199
Underground Infrastructure Capital	31
Tailings and Water Management	27
Mine Equipment Maintenance	18
Underground Electrical	14
Paste Backfill	15
Reclamation / Closure Costs	8
Other	11
Total Sustaining Capital	**325**

1.18.2 Operating Costs

Operating expenses were calculated using the Sugar Zone Mine's 2021 budget as a reference point where applicable and developed from first principles when budgetary items were not available. The 2021 budget costs were adjusted in the successive years to reflect increases related to mining and processing ore at higher rates as well as to incremental increases in mining unit costs as the average depth of mining increases over the LOM. The fixed and variable components of operating costs were analyzed to accurately reflect the expected operating cost profile at higher tonnage throughput.

The total LOM operating costs, summarized in Table 1-9, amount to $704 million, which translates into an average cost of $205/t ore processed over the LOM ($197/t from 2023 to 2027 when the mine is at 1,200 t/d).

Table 1-9: Summary of Operating Costs

		Unit	LOM
Physicals	Ore Mined	t '000s	3,454
	Ore Processed	t '000s	3,454
Costs	Mining	$ '000s	419,890
	Processing	$ '000s	121,965
	Site G&A	$ '000s	162,531
	Total	**$ '000s**	**704,385**
Unit Costs	Mining	$/t	122
	Processing	$/t	36
	Site G&A	$/t	48
	Total	**$/t**	**205**

1.19 Economic Analysis

The Sugar Zone Mine expansion to 1,200 t/d is economically viable and presents an attractive 89% post-tax internal rate of return (IRR) over the existing 800 t/d mine plan and a post-tax net present value at a 5% discount ($NPV_{5\%}$) of $332 million generated from $512 million in mine free cash-flow. The summarized results of this evaluation are shown in Table 1-10.

The economic model developed for the Project is an asset level model and excludes corporate costs and capital structure adjustments.

Table 1-10: Summary of Economic Results of Expansion to 1,200 t/d

	Unit	2021 Throughput: 800 t/d	2023 Throughput: 1,200 t/d	LOM Average 2023 to 2027	LOM Total 2021 to 2029
Macro Assumptions					
Gold Price[1]	US$/oz Au	1,938	1,782	1,677	1,709
Exchange Rate	US$:C$	1.00:0.76	1.00:0.76	1.00:0.76	1.00:0.76
Production					
Tonnes Processed	t '000s	292	438	435	3,454
Grade	g/t Au	7.1	7.7	7.5	7.2
Recovery	%	94.0	94.4	94.4	94.3
Gold Recovered	oz Au	62,458	102,261	98,743	751,601
Cost Metrics					
Cash Cost—Before Royalties[2]	US$/oz Au	823	651	684	737
Cash Cost—Net of Royalties[2]	US$/oz Au	898	721	749	804
Mine Site AISC[2]	US$/oz Au	1,430	1,002	1,025	1,132
Capital					
Mine Development	$ M	26	21	23	199
Sustaining Capital	$ M	18	17	12	126
Expansion Capital[3]	$ M	3	2	-	21
Cash Flow					
Net Revenues[4]	$ M	149	225	204	1,582
Mine EBITDA[5]	$ M	84	140	119	878
Mine Free Cash Flow[6]	$ M	36	96	80	512
NPV and IRR					
Pre-Tax $NPV_{5\%}$	$ M				417
Post-tax $NPV_{5\%}$	$ M				332
IRR vs. 800 t/d[7]					89

Notes:
[1] Analyst consensus gold price estimates.
[2] Cash cost and AISC are non-IFRS measures and reported at the project level. AISC excludes corporate G&A.
[3] Expansion capital is $3 million in 2021, $16 million in 2022, and $2 million in 2023.
[4] Revenues after transportation and refining charges, excludes hedge payments.
[5] Net revenues less production costs, excludes hedge payments, interest, tax, capital, and corporate costs.
[6] Operating cash flow net of capital, tax and NPI, excludes corporate costs and hedge payments.
[7] IRR is calculated on a differential post-tax cash flow basis between 1,200 t/d expansion and operating at 800 t/d.

The Feasibility Study was based on analyst consensus forward price estimates incorporating 31 broker estimates and a fixed US$:C$ conversion rate of 0.76 (Table 1-11).

Table 1-11: Base Case Macroeconomic Assumptions

	Unit	2021	2022	2023	2024	2025+
Gold Price	US$/oz	1,938	1,871	1,782	1,735	1,624
Exchange Rate	US$:C$	1.00:0.76	1.00:0.76	1.00:0.76	1.00:0.76	1.00:0.76

The Feasibility Study returned a pre-tax NPV5% of $417 million and a post-tax NPV5% of $332 million. A post-tax IRR of 89% was calculated based on the return of incremental post-tax cash flows between 1,200 t/d and at 800 t/d throughput.

NPV and IRR are summarized in Table 1-12 using both the base case and flat gold price estimates.

Table 1-12: $NPV_{5\%}$ and IRR at Varying Gold Price

		US$1,600	US$1,700	Base Case	US$1,800	US$1,900
Pre-Tax $NPV_{5\%}$	$ M	329	402	417	476	549
Post-tax $NPV_{5\%}$	$ M	267	321	332	371	420
IRR[1]	%	76	86	89	95	105

Note: IRR calculated on a differential post-tax cash flow between 1,200 t/d expansion and operating at 800 t/d.

The economic model was subjected to a sensitivity analysis to determine the effects of changing metals prices, production costs, capital costs and sustaining capital. The NPV is most sensitive to gold price, Foreign Exchange Rate (FX) and mine costs as summarized in Table 1-13.

Table 1-13: Post-tax $NPV_{5\%}$ Sensitivity ($ million)

	-10%	-5%	Base Case	5%	10%
Gold Price	245	289	332	375	417
US$:C$	426	377	332	291	254
Mine Costs	354	343	332	321	310
Expansion Capital	334	333	332	331	331
Sustaining Capital	340	336	332	328	324

Given existing tax loss pools, the Sugar Zone Mine is expected to pay minimal tax for the next two years (2021 and 2022), after which the Sugar Zone Mine is subject to government taxes based on the following:

- Ontario Mining Tax—10%
- Federal and provincial income tax—25%
- The economic analysis factors in existing tax pools.

Based on the assumed tax rate and pre-tax financial projections in the model, the Sugar Zone Mine is expected to incur mining taxes of $109 million over a nine-year mine life, implying an average effective tax rate of 21%.

Production at the Sugar Zone Mine is subject to two NSR royalties. Each NSR is 2.0%, for a total of 4.0%.

1.20 Interpretations and Conclusions

Harte Gold personnel reviewed and audited the historical exploration data available for the Sugar Zone gold mine, as well as the exploration methodologies adopted to generate the data. Exploration work is professionally managed, and procedures are adopted that meet accepted industry best practices. The author is of the opinion that the exploration data are sufficiently reliable to interpret with confidence the boundaries of the gold mineralization, and they support evaluation and classification of Mineral Resources in accordance with generally accepted *CIM Estimation of Mineral Resource and Mineral Reserve Best Practices Guidelines* and *CIM Definition Standards for Mineral Resources and Mineral Reserves*.

The Project's drilling database includes information from 972 drill holes and 355,593 m, of which 750 drill holes totalling 339,234 m were used to generate the Mineral Resource estimate.

The primary mining method used at the Sugar Zone Mine is longitudinal-retreat longhole open stoping, which is deemed suitable considering the geometry of the orebody.

Harte Gold performs regular gold ounce reconciliations between production and the mineral resource block model, and results have generally been within industry-acceptable ranges, with current results skewing slightly positive. Harte Gold uses the reconciliation process to validate its Mineral Resource estimation parameters and procedures.

Harte Gold has undertaken a detailed engineering and economic study of an expansion of mine and mill throughput from 800 t/d to 1,200 t/d to mine the Mineral Reserves. Harte Gold has concluded that mining the current Mineral Reserves at an increased rate of 1,200 t/d is both feasible and economically beneficial.

The current 800 t/d mill consistently achieves recoveries of 94%. In undertaking the Project, the mill will be expanded to 1,200 t/d with replacement of the secondary cone crusher, addition of a ball mill, modification to the gravity and flotation circuits, and other upgrades, additions, and expansions within the process plant.

As part of the Project, the mining mobile equipment fleet will also be expanded by adding one 30-tonne haul truck, three additional 1.8 m^3 LHDs, and additional ancillary equipment such as personnel carriers.

The existing and planned site infrastructure aside from the process plant expansion is not expected to require any material modifications to enable the expansion, although some facilities such as the South TMF or ventilation raises may need to be constructed sooner than if the Project did not move ahead.

The Sugar Zone Mine is operating within environmental compliance, although a number of operational permits will need to be amended to allow for the expansion. Permitting is expected to take between 12 and 28 months.

The Sugar Zone Mine has been and will continue to be a major contributor to the local economy. Harte Gold will continue to engage and work with area Indigenous communities and other communities of interest.

The Project is economically viable, with a post-tax IRR of 89%, which is calculated on the differential post-tax cash flow between the expansion scenario and the base-case scenario of continuing to mine at 800 t/d. Post-tax $NPV_{5\%}$ is $332 million.

1.21 Recommendations

At the conclusion of this Feasibility Study the following recommendations are being made:

- The Feasibility Study shows positive economic benefits for an expansion of the Sugar Zone Mine to 1,200 t/d throughput and the Project should move forward
- Continue definition drilling programs to upgrade the quality of the known Mineral Reserves and identify extensions within the mining area
- Continue current focus on reconciling mined material versus the Mineral Resource block model to improve understanding of the Mineral Resource
- Continue to improve planning and operational processes with a focus on minimizing unplanned dilution and improving productivities
- Proceed with detailed design and engineering of the process plant expansion along with identification of long lead items
- Perform a detailed trade-off study on paste backfill versus waste rock backfill to evaluate opportunity to delay or remove paste backfill capital and operating expenses
- Contingent on outcome of paste backfill trade-off study proceed with detailed design of paste backfill plant and underground distribution system
- Proceed with detailed design work of South TMF and continue environmental baseline monitoring in the proposed location

2 INTRODUCTION

In a press release dated 20 January 2021, Harte Gold Corp. (Harte Gold or the Company) announced the results of its Feasibility Study for Expansion to 1,200 t/d (the Feasibility Study) on the Sugar Zone Mine (the Operation), located in Northern Ontario, Canada. Having successfully entered commercial production in Q1 2019 with a planned production rate of 800 t/d, the objective of the Feasibility Study was to consider the most cost- and capital-effective strategy to increase annual production for 800 t/d to 1,200 t/d (the Project). This Technical Report also supports the disclosure of an updated Mineral Resource Estimate (effective date 30 September 2021) and an updated Mineral Reserve Estimate (effective date 31 December 2020). This Technical Report outlines the results of the Feasibility Study, and was prepared in accordance with the National Instrument 43-101 (NI 43-101) of the Canadian Securities Administrators.

The Sugar Zone Property (the Property) is 100% owned by the Harte Gold, and is located in the Sault Ste. Marie Mining Division, Ontario. Harte Gold is a public company trading on the Toronto Stock Exchange (TSX), with the trading symbol HRT. The Company has its head office at:

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This Technical Report has an effective date of 1 January 2021.

2.1 Sources of Information

This National Instrument (NI) Technical Report—Feasibility Study for Expansion to 1,200 t/d Harte Gold Sugar Zone Mine, Ontario, Canada (the Technical Report) is based on information collected by the Company and individual Qualified Persons (QP) during site visits, discussion with Harte Gold personnel, public information, and additional information provided by Harte Gold. The QPs have no reason to doubt the reliability of the information provided by the Company. Internal Technical Reports and maps, published government reports, letters, memoranda, public disclosure, and public information are listed in Section 27. Sections from reports authored by other consultants have been directly quoted or summarized in this Technical Report, and are indicated where appropriate.

This Technical Report was prepared by Harte Gold with the services of several consulting firms to design and cost the components making up the various options. These companies are:

- Deswik Consulting (Deswik) was commissioned to assist with the underground mine design and scheduling used to generate the Mineral Reserve estimate.

- Halyard Inc. (Halyard) was commissioned to interpret and expand the metallurgical testwork, evaluate process flowsheet options to expand the existing process plant to 1,200 t/d, and undertake the design and engineering for the chosen flowsheet (Halyard, 2021).
- Paterson & Cooke Canada Inc. (P&C) was commissioned to undertake paste backfill testwork, evaluate the suitability of the existing paste backfill plant, design any modifications to the paste backfill plant, and complete the design of the underground paste backfill distribution system (P&C, 2020).
- Wood Canada Limited (Wood) was commissioned to evaluate the life-of-mine (LOM) tailings requirements, tailings management facilities (TMF), and site-wide water balance, in addition to developing the TMF design (Wood, 2021 & 2021a).
- Jodouin Mine Ventilation Ltd (JMVL) undertook the ventilation network design and engineering (JMVL, 2020).
- Environmental Applications Group (EAG) was engaged to evaluate the environmental, permitting, and social aspects of the Project; they were also responsible for the closure cost estimation and management of work done by sub-consultants in the area of water treatment and mine closure bulkheads.
- Stantec Consulting Services Inc. (Stantec) was engaged to evaluate the underground mine services of compressed air, electricity, process water, and de-watering (Stantec, 2021); they also assisted with the first principles build-up of some of the mining unit costs, and compiling the operating cost model.
- BQE Water (BQE) was commissioned to evaluate the existing effluent treatment plant performance and recommend design modifications, as well as to design the water treatment system for increased water-discharge rates upon commissioning of the South TMF (BQE, 2021).
- Ausenco was engaged to evaluate process plant performance and ore-sorting amenability at a scoping level (Ausenco, 2020).
- All consultants were engaged to provide input, and contributed to the development of the operating cost (OPEX) and capital and sustaining capital expenditures (CAPEX).
- Harte Gold reviewed and developed those elements of the Project relating to the geological setting and mineralization, Mineral Resources, mine plan, Mineral Reserves, market studies and contracts, and economic analysis. Harte Gold compiled the overall Technical Report.

The following summarizes the dates of the QP site visits to the Property:

- Chris McCann, P.Eng., Harte Gold's Director of Technical Services, a QP under the regulations of NI 43-101, completed multiple site visits between February 2020 and December 2020.
- Vincent Cardin-Tremblay, P.Geo., formerly Vice President of Geological Services for Harte Gold, a QP under the regulations of NI 43-101, completed multiple site visits between July 2020 and November 2020.

- Justin Taylor, P.Eng., Principal, President, and Director of Halyard, a QP under the regulations of NI 43-101, completed multiple site visits between January 2017 and October 2020, with the most recent being 22 October 2020. Mr. Taylor led the initial design and construction of the Property's process plant. Justin's two site visits in 2020 to evaluate the operating conditions of the process plant were directly related to his involvement in the Feasibility Study.
- Ivana Sabaj Abumohor, P.Eng., Lead Process Engineer for Halyard, a QP under the regulations of NI 43-101, completed multiple site visits between October 2018 and February 2019, with the most recent being 28 Feb 2019. Ms. Sabaj Abumohor was involved with commissioning the Property's process plant. Ms. Sabaj Abumohor did not complete a site visit in relation to Feasibility Study, but relied on two site visits by Halyard employees Justin Taylor, P.Eng., and Dion Deetlefs.
- Rob Brown, P.Eng., of P&C, a QP under the regulations of NI 43-101, did not conduct a site visit, but relied on a site visit by P&C staff. Mr. Mauritz van der Walt visited the site from 13 August 2020 to 14 August 2020. The purpose of the site visit was to audit the existing paste backfill plant, assess the performance of the mechanical equipment, and collect material samples for further testwork.
- Ganan Nadarajah, P.Eng., of Wood, a QP for tailings management facilities and water balance studies, has completed three site visits between 2017 and 2020, with the most recent from 17 August 2020 to 18 August 2020. Mr. Nadarajah is the Engineer of Record (EoR) for the existing North TMF structure, and one site visit was to inspect the North TMF for the annual dam safety inspection (DSI) and evaluate the site of the proposed South TMF location.
- Ron de Gagne, P.Geo., of Environmental Applications Group (EAG), a QP under the regulations of NI 43-101, did not conduct a site visit, but relied on site visits by EAG staff. Mr. Darryl Boyd has visited the site multiple times between 2010 and 2020, with the most recent in February 2020; Mr. Boyd has been involved in many aspects of permitting and environmental monitoring of the Property since early exploration stages.

Table 2-1 presents the authors and co-authors of each section of this Technical Report, who, acting as QP as defined by NI 43-101, take responsibility for those sections of the Technical Report as outlined in the Certificate of Author included in Section 28.

Table 2-1: Summary of Qualified Persons

Section	Description	Qualified Person	Company	Co-author(s)
1	Summary	All in part		
2	Introduction	Chris McCann, P.Eng.	Harte Gold	
3	Reliance on Other Experts	Chris McCann, P.Eng.	Harte Gold	
4	Property Description and Location	Chris McCann, P.Eng.	Harte Gold	
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Chris McCann, P.Eng.	Harte Gold	
6	History	Chris McCann, P.Eng.	Harte Gold	
7	Geological Setting and Mineralization	Vincent Cardin-Tremblay, P.Geo.	Harte Gold	
8	Deposit Types	Vincent Cardin-Tremblay, P.Geo.	Harte Gold	
9	Exploration	Vincent Cardin-Tremblay, P.Geo.	Harte Gold	
10	Drilling	Vincent Cardin-Tremblay, P.Geo.	Harte Gold	
11	Sample Preparation, Analysis and Security	Vincent Cardin-Tremblay, P.Geo.	Harte Gold	
12	Data Verification	Vincent Cardin-Tremblay, P.Geo.	Harte Gold	
13	Mineral Processing and Metallurgical Testing	Ivana Sabaj Abumohor, P.Eng.	Halyard	Justin Taylor, P Eng.
14	Mineral Resource Estimates	Vincent Cardin-Tremblay, P.Geo.	Harte Gold	
15	Mineral Reserve Estimates	Chris McCann, P.Eng.	Harte Gold	
16	Mining Methods	Chris McCann, P.Eng.	Harte Gold	Rob Brown, P.Eng. (P&C)
17	Recovery Methods	Justin Taylor, P.Eng.	Halyard	Ivana Sabaj Abumohor, P Eng.
18	Project Infrastructure	Chris McCann, P.Eng.	Harte Gold	Ganan Nadarajah, P.Eng. (Wood)
19	Market Studies and Contracts	Chris McCann, P.Eng.	Harte Gold	
20	Environmental Studies, Permitting and Social or Community Impact	Ron deGagne, P.Geo.	EAG	
21	CAPEX and OPEX Costs	Chris McCann, P.Eng.	Harte Gold	
22	Economic Analysis	Chris McCann, P.Eng.	Harte Gold	
23	Adjacent Properties	Chris McCann, P.Eng.	Harte Gold	
24	Other Relevant Data and Information	N/A		
25	Interpretations and Conclusions	All in part		
26	Recommendations	All in part		
27	References	All in part		

2.2 Units of Measure, Acronyms, Currency, and Rounding

This Technical Report uses the International System of Units (metric system) except for troy units used in the mining industry: for example, the troy ounce (oz), used to report the mass of precious metals.

A glossary containing a comprehensive list of acronyms and units of measure is included table of contents.

All dollar figures are quoted in Canadian dollars ($) unless otherwise noted.

This Technical Report includes information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding, and consequently introduce a small margin of error. Where these occur, the QPs do not consider them to be material.

3 RELIANCE ON OTHER EXPERTS

The QPs have relied on the input from Harte Gold and qualified independent consulting companies in preparing this Technical Report. This report was prepared using the reports and documents noted in Section 27.

The QP's responsibilities were to assure that this NI 43-101 Technical Report met the stipulated guidelines and standards, considering that the designs, engineering, and costing in the report were contributed by multiple consultants and internal Harte Gold documents as identified in Section 2.

The information, conclusions, opinions, and estimates contained herein are also based on data, reports, and other information supplied by Harte Gold and the other party sources.

All statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the effective date of this Technical Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Sugar Zone Property is 30 km northeast of the town of White River in northern Ontario. White River is on the route of Ontario Highway 17 (part of the Trans-Canada Highway), approximately midway between Thunder Bay (315 km west) and Sault Ste. Marie (260 km southeast). The Operation is approximately 715 km northwest of Toronto and 68 km east of the Hemlo Gold Camp.

The Sugar Zone deposit is centred on lat. 48°47’53”N, long. 85°00’36”W (Universal Transverse Mercator 16U 646140 5406900 [NAD83]). The overall Property encompasses National Topographic System Zones 42C/10, 42C/11, 42C/14, and 42C/15; this covers portions of Odlum, Strickland, Gourlay, Tedder, Abraham Bayfield, Cooper, Johns, Nameigos, Mosambik, and Hambleton Townships, and falls within the Sault Ste. Marie Mining Division (Figure 4-1).



Source: Harte Gold, 2018.

Figure 4-1: Location Map

4.2 Property Description and Tenure

The Sugar Zone Property comprises 81,287 ha (813 km^2) and consists of the Harte Gold leases (surface and mining rights), Harte Gold claims and the Halverson Option (Figure 4-2). The Harte Gold leases cover 1,467 ha (14.7 km^2). The Harte Gold claims comprise 197 multi-cell claims, 43 single-cell claims, and 69 boundary-cell claims. Harte Gold owns 100% of the Harte Gold leases and the claims. The Halverson Option is subject to an option agreement whereby Harte Gold may acquire a 100% interest in the Halverson Option, which comprises four single-cell claims and 12 boundary-cell claims.

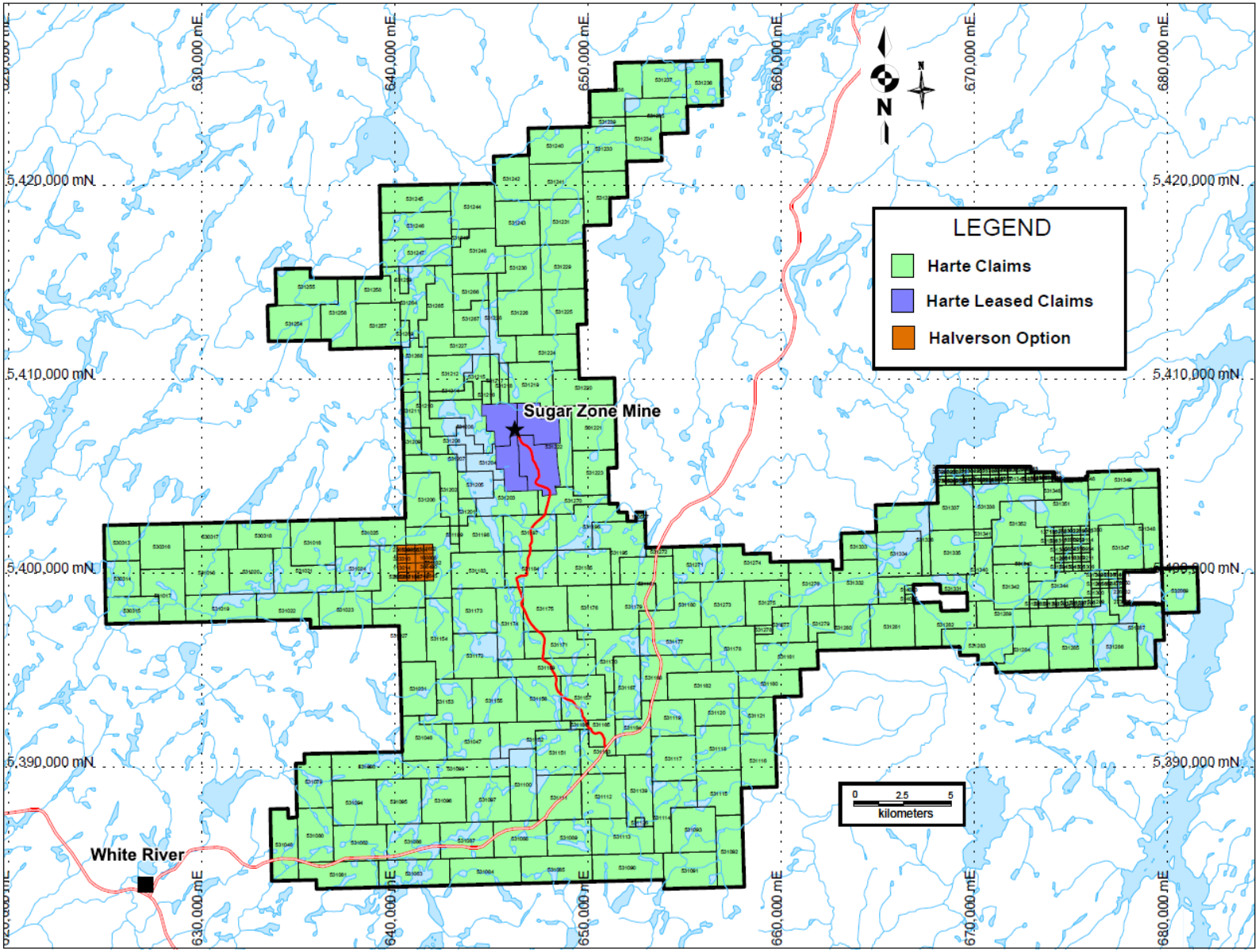
Historically, the names for the Sugar Zone Property, "Dayohessarah Lake," "Dayohessarah," "Dayo," and "Sugar Zone," have been used interchangeably. Dayohessarah Lake is a large body of water occupying the centre of the Property, while Sugar Zone refers to the sugary nature of the quartz veining that hosts gold mineralization on the Property.

On 31 January 1995, Harte Gold acquired the majority of the unpatented mining claims comprising the Sugar Zone Property from John E. Ternowesky, Lloyd Halverson, Ernie Beaven, Eino Ranta, Omer L. Belisle, and Peter Nabigon. Harte Gold subsequently entered into an Option and Joint Venture Agreement with Corona Gold Corporation (Corona) dated 10 July 1998 (the Sugar Zone Joint Venture). Under the Sugar Zone Joint Venture, Corona was granted the right to acquire a 51% interest in, and become the Operator of, the Property upon the payment on closing of $50,000 to Harte Gold and payment of $1 million in exploration expenditures within 48 months of the Option and Joint Venture Agreement; Corona exercised that right. Corona also had the right to increase its interest in the Property to 75% on payment of an additional $200,000 to Harte Gold, and an additional $700,000 in exploration expenditures within 48 months of the date of the Sugar Zone Joint Venture.

Pursuant to a Letter Agreement dated 5 March 2010 between Harte Gold and Corona, the parties entered into an Option Agreement (the Corona Option) dated 28 May 2010, entitling Harte Gold to acquire Corona's 51% interest in the Sugar Zone Joint Venture on completion of certain conditions, including:

- An initial cash payment to Corona of $10,000 and 7,180,000 Harte Gold common shares, whereupon Harte Gold would become the operator of all activities on the Property
- A further cash payment to Corona of $2,000,000, and the issuance of an additional 4,331,638 common shares to Corona
- Payment of $90,000 in cash on or before the six-month anniversary of the Corona Option
- Payment of $2,500,000 on or before the second anniversary of the Corona Option, or $3,000,000 on or before the third anniversary of the Corona Option.

Effective 10 March 2010, Harte Gold became the Operator of the Joint Venture and on 23 May 2012, Harte Gold made the payment of $2,500,000 to Corona, thus becoming the 100% owner of the Property.



Source: Harte Gold, 2019.

Figure 4-2: Sugar Zone Property Claim Map

The original 313 claims were subject to a 3.5% net smelter royalty (NSR), of which Harte Gold had the option to acquire 1.5% for $1.5 million; Harte Gold exercised its option and bought down 1.5% of the NSR, effective 31 October 2018 (Harte Gold, 2018, November 15). Harte Gold also has the right of first refusal on the remaining 2.0% NSR.

On 28 June 2010, Harte Gold entered into an option agreement to acquire three mining claims situated in the central part of the Sugar Zone Property (the Halverson claims: SSM 4228496, SSM 4228497, and SSM 4228499) from vendors Lloyd Halverson, Eugene Belisle, and John E. Ternowesky. The agreement states that for Harte Gold to earn a 100% interest in the claims it must:

- Make cash payments of $225,000
- Incur work commitments of $300,000 over five years (2010–2015)
- Issue 200,000 common shares over three years (2010–2013), subject to a 3% NSR, which could be reduced to 1.5% on payment of $1,500,000.

In addition to the above, if, after the five-year option periodan economically viable deposit were to be found on the claims—as defined by an independent Feasibility Study—Harte Gold would make annual payments of $20,000 against future NSR payments. In the event that an economically viable deposit was not found, Harte Gold would make annual payments of $20,000 for a period of five years to complete its purchase of the claims. The mining claims were transferred to Harte Gold effective 19 June 2014.

In November 2010, 83 unpatented mining claims were staked adjacent to the Sugar Zone Property to provide a buffer around the mining claims that were considered to hold the highest potential to convert into an economically feasible deposit.

In May and June 2015, the Company was granted 21-year leases on four mining claims comprising 77 of the formerly unpatented, unsurveyed, contiguous, mining claims. In 2017, the Sugar Zone Property was further extended by the addition of 208 staked claims covering an additional 48,640 ha (486 km^2).

In August 2017, four unpatented mining claims (Halverson Option) were optioned from Lloyd Halverson, Doug Kakeeway, and John E. Ternowesky, subject to a Property Option Agreement whereby Harte Gold may acquire a 100% interest in the Sugar Zone Property subject to a 3% NSR by paying $32,000, issuing 300,00 common shares, performing an airborne survey of the Sugar Zone Property and completing a drill program of a least 400 m.

The four Sugar Zone Property leases—CLM514, CLM515, CLM516, and CLM517—cover an area of 1,467 ha (14.7 km^2). The leases were issued in 2015 and are valid to 2036, subject to payment of property taxes.

The Sugar Zone Property is subject to the following NSRs:

- A 1.5% NSR held by 2729992 Ontario Corp. (an affiliate of ANR Investments B.V.) granted on 18 December 2019 (Appian 2019 Royalty)
- A 0.5% NSR held by 2729992 Ontario Corp. (an affiliate of ANR Investments B.V.) granted on 17 July 2020 (Appian 2020 Royalty)
- A 2.0% NSR held by Lloyd Halverson, John E. Ternowesky, Ernie Beaven, Eino Ranta, the Estate of Omer L. Belisle, Broad Horizons Trust and Broad Horizons Inc. granted on 10 July 1998 (Corona/Harte JV Royalty).

Details of claim recording dates, due dates, work requirements, and work credits are shown in Appendix A. The Sugar Zone Property staked claims are on Crown Land and comprise mineral rights only.

The claims information presented in this section is valid as of the effective date of this Technical Report. In 2018, the Ontario Ministry of Northern Development and Mines (MNDM) converted from a system of ground staking to online registration of mining claims. The functionality of the new system is being improved over time.

Ontario Crown lands are available to licensed prospectors for the purpose of mineral exploration. A licensed prospector must first stake a mining claim to gain the exclusive right to explore on Crown land. Claim staking is governed by the Ontario *Mining Act* and is administered through the Provincial Mining Recorder and Mining Lands offices of the MNDM.

A claim remains valid as long as the claim holder properly completes and files the assessment work as required by the *Mining Act*, and the Minister approves the assessment work. A claim holder is not required to complete any assessment work within the first year of recording a mining claim. Immediately following the initial staking date, the claim holder has two years to file one year's worth of assessment work; to keep a mining claim current, the holder must perform $400 worth of approved assessment work per mining claim unit, per year. Claims are forfeited if the assessment work is not done.

A claimholder may prospect or carry out mineral exploration on the land under the claim; however, the land covered by these claims must be converted to leases before any development work or mining can be performed. Mining leases are issued for 21-year terms and may be renewed for 21-year periods. Leases can be issued for surface and mining rights, mining rights only, or surface rights only. Once issued, the lessee pays an annual rent to the province. Furthermore, prior to bringing a mine into production, the lessee must comply with all applicable federal and provincial legislation.

4.3 Environmental and Permitting

Prior to Harte Gold's bulk sample program there had been no previous mine production on the Sugar Zone Property. Reclamation of the Sugar Zone Property resulting from the bulk sample mining is covered under an approved Closure Plan that Harte Gold filed with the MNDM.

Harte Gold currently has 18 work permits covering different areas of the Sugar Zone Property, which allow Harte Gold to conduct exploration activities such as line-cutting, geophysical and geochemical surveys, as well as diamond drilling Table 4-1). Furthermore, Harte Gold recently submitted three additional work-permit applications to the Ministry for exploration activities to be performed in the East Claim area. Harte Gold also has an active work permit for the Stoughton–Abitibi property located east of Matheson, Ontario.

Table 4-1: Harte Gold Claims and Leases

	Exploration Permit #	Permit Name	Date Submitted	Issue Date	Expiry Date
1	PR-17-11209	Eagle Zone	-	12-Sep-18	12-Sep-21
2	PR-18-11273	Highway Zone	-	12-Sep-18	12-Sep-21
3	PR-18-000290	Nameigos Area 6	-	10-May-19	10-May-22
4	PR-18-000291	K7 South Areas 2-4	-	10-May-19	10-May-22
5	PR-18-000292	K7 South Area 1	-	10-May-19	10-May-22
6	PR-18-000297	Hambleton E, W, N Grids	-	10-May-19	10-May-22
7	PR-18-000298	Hambleton Area 1	-	10-May-19	10-May-22
8	PR-18-000299	Hambleton Areas 2-4	-	10-May-19	10-May-22
9	PR-18-000300	Hambleton Areas 5-8	-	10-May-19	10-May-22
10	PR-18-000301	Nameigos Areas 1-5	-	10-May-19	10-May-22
11	PR-18-000302	Nameigos Area 1a	-	10-May-19	10-May-22
12	PR-19-000023	Eagle West Areas 1-2	-	10-May-19	10-May-22
13	PR-19-000024	Eagle West Area 3	-	10-May-19	10-May-22
14	PR-19-000198	TNT Area	-	20-Sep-19	19-Sep-22
15	PR-19-000312	TT8 Zone	-	24-Jan-20	24-Jan-23
16	PR-20-000030	Near Mine - A	-	24-Mar-20	24-Mar-23
17	PR-20-000031	Near Mine - B	-	24-Mar-20	24-Mar-23
18	PR-20-000067	Stoughton Abitibi	-	03-Jun-20	03-Jun-23
19	PR-20-000372	Southern Zone	-	11-Feb-21	10-Feb-24
20	Application Submitted	Nameigos East	12-Feb-21	-	-
21	Application Submitted	Nameigos South	12-Feb-21	-	-
22	Application Submitted	Nameigos West	13-Feb-21	-	-

Source: Harte Gold, 2021.

Areas surrounding Dayohessarah and Hambleton Lakes are designated by the Ontario Ministry of Natural Resources as “Restricted Access.” Locked gates on Road 200 and Road 305 control vehicular access to prevent access to remote lodge operations on the two lakes. Permits are required for road access to most of the Sugar Zone Property for mineral exploration purposes. Harte Gold has entered into an agreement with the remote lodge operator (the Access Agreement) that provides a framework for access by Harte Gold and its employees, contractors, and others associated with the exploration and development of the Sugar Zone Property. The Access Agreement is in effect through closure of mining activities on the Sugar Zone Property, subject to early termination by Harte Gold should the Company cease exploration or mining operations, or otherwise abandon the Sugar Zone Property.

In 2015, Harte Gold received all regulatory and permitting approvals required to proceed with a 70,000-tonne bulk sample program of the Sugar Zone deposit. The bulk sample project began in Q4

2015 and was completed in March 2017. In January 2017, Harte Gold received approvals required to expand its mining operation with a Phase 1 commercial production program for an additional 30,000 tonnes. In May 2017, the Company obtained approvals to construct a process plant. Harte Gold completed and signed an Impact Benefits Agreement (IBA) with the Pic Mobert First Nation (PMFN), which is proximal to the Operation (Harte Gold, 2018, May 2). The PMFN and Pic River First Nations are situated in the same watershed as the Sugar Zone Property and have confirmed no outstanding concerns related to the Operation.

Final operating permits were received from the Government of Ontario to start commercial production at the Sugar Zone Mine (Harte Gold, 2018, September 20) and Harte Gold declared commercial production at the Sugar Zone Mine effective January 1, 2019 (Harte Gold, 2019, January 8).

There is no environmental litigation or pending fines associated with the Sugar Zone Property.

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

The Sugar Zone Operation is located 30 km north of Ontario Highway 17 (Trans-Canada Highway). From there, it is readily accessed via Highway 631, a secondary paved thoroughfare extending north from White River and passing approximately 10 km east of the Sugar Zone Property. A 20 km long gravel haulage road leads west from Highway 631 to the Property. The distance from White River to the Sugar Zone Property is approximately 60 km by road.

The town of White River has a population of 645 (2016 Census). The cities of Sault Ste. Marie (population 73,368) and Thunder Bay (population 108,359) are 311 km south and 383 km west of White River, respectively, along Highway 17. Highway 631 passes through Hornepayne (population 1,050) on the way to Highway 11.

Access is also available by float plane, based in White River, via Dayohessarah Lake or Hambleton Lake, and by helicopter, based in Wawa or Marathon.

The western and southern portions of the Sugar Zone Property are accessible via a series of logging roads controlled by White River Forest Products Limited. Road 100 extends north from the western end of White River. Road 200 intersects Road 100 20 km from Highway 17 and provides access to the western and southern portions of the Sugar Zone Property. Road 300 intersects Road 100 36 km from Highway 17 and provides access to the very northern portion of the Sugar Zone Property. Road 305 intersects Road 300 6 km from Road 100 and provides access to northern and eastern parts of the Sugar Zone Property. Figure 5-1 displays all referenced routes and key points of interest around the Sugar Zone Property.

5.1 Climate

The Operation area is characterized by a northern boreal climate, with short hot summers and cold, snowy winters. The temperature ranges from −35°C in the winter to +30°C in the summer; mean temperatures are approximately −21°C to +20°C. Average annual rainfall is 727 mm, with the wettest month being September (120 mm average). Snow is abundant, often reaching several metres of accumulation over the season, with December and January having the heaviest snowfalls, seeing an average of approximately 80 cm of snowfall in each month. Snow is on the ground by late October, and the ice begins to thaw on the lakes by April. Exploration and mining activities can generally be carried out year-round.

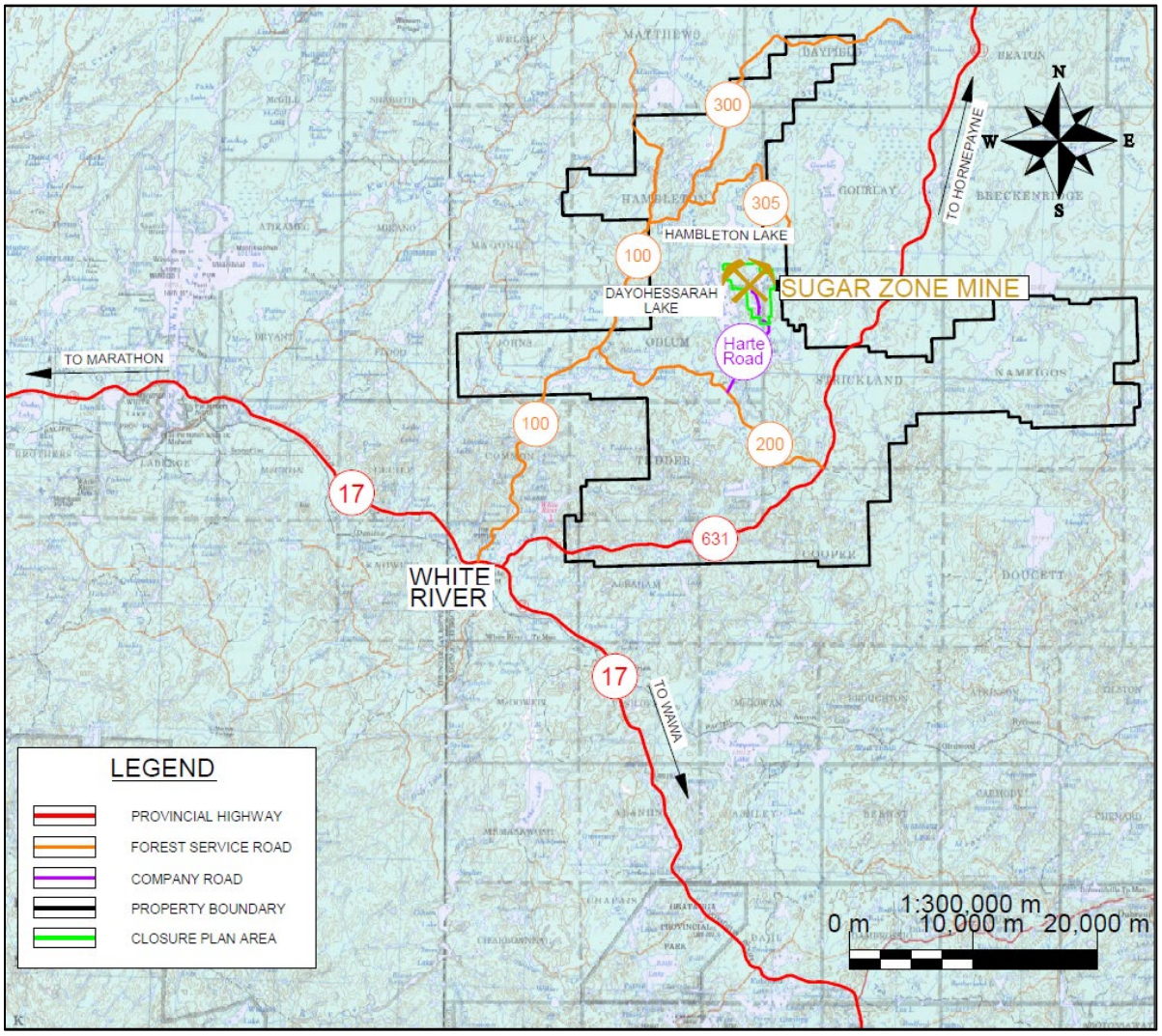


Figure 5-1: Sugar Zone Mine and Surrounding Area with Provincial and Local Roads

5.2 Infrastructure

A Hydro One 25 kV electrical transmission line has been extended from White River to the mine site. The power supply to site is stable and experiences only occasional interruptions from the White River transformer. Future improvements to power supply stability are expected upon the early 2022 completion of the East-West Tie Transmission Line Project being undertaken by NextBridge Infrastructure. This project is a major upgrade to the transmission capacity between the Wawa Transformer Station and the Lakehead Transformer Station (near Thunder Bay).

The Canadian Pacific transcontinental main rail line passes through White River, and the Canadian National Railway main line passes through Hornepayne (approximately 65 km north of White River).

Hornepayne has a paved airstrip that can accommodate charter flights.

The Operation benefits from close proximity to the Hemlo Mining Camp 68 km west, and mining operations in the Wawa area, 90 km to the south. Mining infrastructure and competent workers are present in the communities of Marathon and Manitouwadge, serving the Hemlo mining camp, as well as Wawa. A number of businesses provide services to the exploration sector, and a long history of mining in the region contributes to a highly skilled work force. The main commercial centres for the region are Sault Ste. Marie and Thunder Bay.

The Sugar Zone Property is serviced by existing on-site offices. Harte Gold maintains accommodations for employees in White River. Abundant water sources are present in the lakes, rivers, and creeks throughout the area. There is sufficient space on the Sugar Zone Property to build and extend mining infrastructure.

5.3 Physiography

The Property is in the Canadian Shield, where relief varies from moderate to rugged. Dayohessarah Lake, west of the Sugar Zone, has an elevation of 394 masl, with surrounding ridges reaching 480 masl.

The overburden, as observed in trenching and diamond drilling, is typically up to 10 m thick across the Sugar Zone Property, with up to 3 m of overburden on the Sugar Zone. Rock outcrops are relatively abundant, especially on higher ridges. The central portion of the Property, which contains the Sugar Zone mineralization, exhibits 10% to 15% bedrock exposure, while the overall Sugar Zone Property has an estimated 5% to 10% outcrop exposure. The entire area has been covered with a thin, discontinuous veneer of glacial till and outwash material.

Vegetation is characteristic of the boreal forest, with jack pine, fir, poplar, and birch occupying dry uplands, and cedar, tamarack, and spruce on more poorly drained terrain. Alders and Labrador tea predominate in lower-lying, swampy areas.

The Sugar Zone Property lies close to the ‘height of land,’ or Laurentian Continental Divide, that marks the boundary between the Great Lakes and Arctic watersheds. Dayohessarah Lake and the area around the Sugar Zone Property drain north into the Shabotik River, then west and south into White Lake and the White River, which flows into Lake Superior southeast of Marathon.

6 HISTORY

This section outlines exploration conducted between the 1960s and 2010, when Harte Gold initiated work on the Property. The information on historical exploration is mainly summarized from Hunt's (2010) NI 43-101 Technical Report. The information was primarily obtained from the MNDM assessment files in the Resident Geologist's Office in Sault Ste. Marie. The majority of the work has been done in the vicinity of Dayohessarah Lake and the Sugar Zone area and follows discovery of the Hemlo Gold Deposit in 1981. This description of historical exploration is focussed on this core area of the Property.

6.1 Exploration History

Exploration for gold and base metals has been performed on the Property since 1969; Table 6-1 summarizes historical exploration up to the beginning of Harte Gold drilling.

Table 6-1: Summary of Exploration History of the Sugar Zone Property Area (1969–2010)

Year	Company/Person	Exploration
1969	Canex Aerial Exploration Ltd.	Drilled three diamond drill holes in the vicinity of the mafic/ultramafic rocks near the north end of Dayohessarah Lake. Results included an intersection of 0.326% Ni and 0.08% Cu over 1.5 m (5 feet) in metagabbroic rocks.
1983–1986	Pezamerica Resources Limited	Airborne Mag and EM survey outlined numerous geophysical anomalies in the area. Teck Exploration drilled nine airborne geophysical targets based on coincidental gold-in-soil anomaly trends. The airborne anomalies were explained by pyrite/pyrrhotite-rich horizons in felsic metavolcanic rocks. Hole PZ 6 returned 0.47% Zn over 0.9 m (2.8 feet); however, no significant gold values were obtained.
1990	Prospecting Syndicate	Staked most of the Dayohessarah greenstone belt.
1991	Hemlo Gold Mines Inc.	Options Property from prospectors' syndicate. Sugar Zone discovered by surface prospecting. Based on bedrock exposure and trenching, the Sugar Zone was traced for 750 m, and a ground induced polarization (IP) survey outlined the Sugar Zone structure over 1,500 m.
1993	Hemlo Gold Mines Inc.	Sugar Zone tested by diamond drilling with six holes totalling 800 m. All drill holes intersected significant gold mineralization. A small trenching program was also initiated on the Sugar Zone. A grid was cut with a 6 km baseline and tie-lines ranging in spacing between 100 m and 1,000 m.
1994	Hemlo Gold Mines Inc.	Fifteen diamond drill holes were completed totalling 2,416 m. Eight drill holes intersected the Sugar Zone. Induced polarization survey (IP), magnetic survey, prospecting, and geological mapping were completed on the grid. Property was returned to prospecting syndicate.
1998	Corona Gold Corporation (51%) and Harte Gold Corp. (49%) Joint Venture	Corona–Harte Gold JV options Property.
1998–1999	Corona–Harte Gold JV	53 NQ diamond drill holes totalling 9,922 m cover 3 km strike length on Sugar Zone at 50 m spacing and test IP anomalies west of Sugar Zone and east of Dayohessarah Lake. Power trenching on Sugar Zone, with six trenches excavated, mapped, and sampled. 96 km of grid with lines at 100 m cut on Sugar Zone with mapping, rock, and soil sampling. Detailed magnetic survey, VLF, and reconnaissance gradient IP survey were completed.
2003–2004	Corona–Harte Gold JV	Step-out and infill diamond drilling on the Sugar Zone with 26 holes for a total of 7,100 m.

Year	Company/Person	Exploration
2004	Corona–Harte Gold JV	Further step-out and infill diamond drilling on the Sugar Zone with 12 holes for a total of 3,563 m.
2008	Corona–Harte Gold JV	A 1,917-line km DIGHEM/mag airborne survey flown by Fugro Airborne Surveys Corp.
2009	Corona–Harte Gold JV	Diamond drilling program totalling 2,007 m in 10 holes. Program tests airborne electromagnetic and magnetic anomalies, plus IP chargeability anomalies and geologically defined targets to the north and the south of the known Sugar Zone mineralization. Prospecting, reconnaissance geological mapping, and channel sampling program completed to evaluate Fugro airborne anomalies. Highlights included sampling of a float rock returning a value of 87.80 g Au/t, as well as grab samples from quartz veining east of the Sugar Zone returning values of 30.40 and 9.04 g/t Au.
2010	Harte Gold Corp.	First Harte drilling program. 12 holes totalling 2,097 m.

Information on the Hemlo Gold Mines Inc. (Hemlo Gold) and Corona–Harte Gold JV programs in the following paragraphs is drawn from Hunt (2010) and Kociumbas and Power-Fardy (2012).

Hemlo Gold drilled six diamond drill holes targeting the Sugar Zone between 17 and 25 September 1993. Hemlo Gold used Chibougamau Diamond Drilling Ltd. to complete 15 more NQ-size diamond drill holes between September and October 1994, of which nine targeted the Sugar Zone. The diamond drill core from within the Sugar Zone is being stored at the Harte Gold core logging facility in White River, and the rest of the core is stored on pallets along Road 305, north of the Sugar Zone.

All of the Hemlo Gold core samples were sent to Chemex Laboratories Ltd., which became ALS Chemex Laboratories Ltd., and more recently known as ALS Minerals Ltd. (ALS), in Vancouver, BC. All samples were assayed for gold with a fire assay using lead collection and an atomic absorption spectrometry (AAS) finish. All drill-hole collars were spotted in reference to the nearest picket on a cut grid with 100 m spaced lines oriented at 50°. Kociumbas and Power-Fardy (2012) report that there is no information on how, or if, downhole surveys were completed. In December 2011 Harte Gold personnel used a Trimble 3000 GeoXT, with sub-metre accuracy, to locate and record the drill-hole collars.

Corona drilled 101 NQ diamond drill holes during three diamond drill program phases between 1998 and 2009. Chibougamau Diamond Drilling carried out all drilling. David S. Hunt, P.Geo., of Sharpstone Geoservices Ltd. (Sharpstone) in Thunder Bay, carried out most field supervision and logging for all three diamond drill programs. The supervising geologist oriented the drill using a Brunton compass, and the drillers carried out downhole surveys at 50 m intervals using a REFLEX EZ-SHOT single-shot unit. All diamond drill core from the 2009 program is currently stored at the core logging facility in White River. All core samples were sent to Accurassay Laboratories in Thunder Bay. The diamond drill core prior to 2009 from the within the Sugar Zone is being stored at the Harte Gold drill-core logging facility in White River. The remainder of the core is stored on pallets along Road 305, north of the Sugar Zone.

All drill-hole collars were spotted in reference to the nearest grid line picket. In December 2011 Harte Gold personnel used a Trimble 3000 GeoXT to locate and record the drill-hole collars.

6.2 Previous Mineral Resource Estimates

Sharpstone Geoservices Ltd.'s David Hunt, P.Geo., prepared an initial Mineral Resource estimate of the Sugar Zone following completion of the 1998 drilling program. The initial Inferred Mineral Resource for the Upper and Lower Subzones was estimated at 429,996 tonnes, with an average grade of 11.19 g/t Au, using a 3.0 g/t Au cut-off grade. Hunt updated the Mineral Resource estimate for the Sugar Zone after the 2003–2004, 2004, and 2010 drilling programs. Hunt's (2010) updated NI 43-101 Mineral Resource for the Upper and Lower Subzones was estimated at an Indicated Mineral Resource of 1.117 Mt grading 8.41 g/t Au, and Inferred Mineral Resource of 0.417 Mt grading 7.13 g/t Au. Hunt's (2010) and previous Mineral Resource estimates were made using the polygonal method, and based on length weighted-average grades using a cut-off grade of 3.0 g/t Au over a minimum estimated true width of 1.45 m, and a bulk density of 2.62 t/m^3.

Watts, Griffis and McOuat Limited (WGM), Toronto, prepared an updated NI 43-101 Mineral Resource estimate (Kociumbus & Power-Fardy, 2012) using a block model method and inverse distance cubed (ID3) weighting for the grade interpolation, employing 0.5 m equal length (normalized) composites for the Upper and Lower Subzones of the Sugar Zone, which had sufficient data to allow for continuity of geology and grades. Table 6-2 summarizes the classified Mineral Resource estimate based on a minimum horizontal width of 1.5 m, a uniform bulk density of 2.62 t/m^3, and a 3.0 g/t Au cut-off grade, using a gold price of US$1,100/oz. The effective date of the WGM Mineral Resource estimate was 27 February 2012.

P&E Mining Consultants Inc. (P&E) (P&E, 2018) prepared an updated NI 43-101 Mineral Resource estimate using a block modelling method and multiple indicator kriging for the grade interpolation, using 1.5 m equal-length (normalized) composites for the Sugar (Upper and Lower Subzones) and Middle Zones that had sufficient data to allow for continuity of geology and grades. Table 6-3 summarizes the categorized Mineral Resource estimate based on a minimum horizontal width of 1.5 m, a uniform bulk density of 2.73 t/m^3, and a 3 g/t Au cut-off grade, using a gold price of US$1,250/oz. The effective date of the P&E Mineral Resource estimate was 15 February 2018.

P&E (P&E, 2019) prepared an updated NI 43-101 Mineral Resource estimate using a block modelling method and multiple indicator kriging for the grade interpolation, using 1.0 m equal length (normalized) composites for the Sugar, Middle, and Wolf Zones that had sufficient data to allow for continuity of geology and grades. Table 6-4 summarizes the categorized Mineral Resource estimate based on a minimum horizontal width of 1.5 m, a uniform bulk density of 2.73 t/m^3, and a 3 g/t Au cut-off grade, using a gold price of US$1,250/oz. The effective date of the P&E Mineral Resource estimate was 14 February 2019.

Table 6-2: WGM (2012) Sugar Zone Mineral Resource Estimate at a 3.0 g/t Au Cut-off Grade

Zone / Classification	Tonnes	Uncapped		Capped	
		Au (g/t)	Au (oz)	Au (g/t)	Au (oz)
Upper Subzone					
Indicated	240,400	6.94	53,600	6.31	48,800
Inferred	38,700	4.65	5,800	4.56	5,700
Lower Subzone					
Indicated	740,500	11.16	265,700	9.50	226,200
Inferred	541,800	8.62	150,200	7.21	125,600
Total Mineral Resources					
Indicated	980,900	10.13	319,300	8.72	275,000
Inferred	580,500	8.36	156,000	7.03	131,300

Notes: Interpretation of the mineralized zones was created as 3-D wireframes/solids based on a 0.5 g Au/t cut-off grade.
Mineral Resources were estimated using a block model with a block size of 1 m x 3 m x 1 m.
Grade capping was done on 0.5 m composited assays; Upper Subzone was capped at 30 g/t Au, and Lower Subzone was capped at 50 g/t Au. Tonnages and grades reported above are undiluted.
Assumed gold price was US$1,100/oz.
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues.
The quantity and grade of reported Inferred Mineral Resources in this estimation are uncertain in nature, and there has been insufficient exploration to define these Inferred Mineral Resources as an Indicated or Measured Mineral Resource; it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource classification.
The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) *Standards for Mineral Resources and Reserves, Definitions and Guidelines*, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council December 11, 2005.

Table 6-3: P&E Mineral Resource Estimate (2018) at 3.0 g/t Au Cut-off Grade

Classification	Zone	Tonnes	Au (g/t)	Au (oz)
Indicated	Sugar (Upper & Lower Subzones)	2,148,000	8.61	594,700
	Middle	460,000	8.09	119,500
	Total	**2,607,000**	**8.52**	**714,200**
Inferred	Sugar (Upper & Lower Subzones)	1,802,000	6.37	369,300
	Middle	1,788,000	6.81	391,500
	Total	**3,590,000**	**6.59**	**760,800**

Notes: Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), *CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines* prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council May 19, 2014.
Bulk sample mined-out area was depleted from the Mineral Resource model.

Table 6-4: P&E Mineral Resource Estimate (2019) at 3.0 g/t Au Cut-off Grade

Classification	Zone	Tonnes (t '000s)	Au (g/t)	Au (oz '000s)
Indicated	Sugar	2,545	8.63	707
	Middle	1,489	7.62	365
	Wolf	209	5.42	36
	Total	**4,243**	**8.12**	**1,108**
Inferred	Sugar	1,474	5.78	274
	Middle	1,246	6.01	241
	Wolf	234	5.86	44
	Total	**2,954**	**5.88**	**559**

Notes: Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource, and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), *CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines* prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council May 19, 2014.
Advanced exploration bulk sample and commercial mined-out areas were depleted from the Mineral Resource model.

All previous Mineral Resource estimates are superseded by the current Mineral Resource estimate in Section 14 of this Technical Report.

7 GEOLOGICAL SETTING AND MINERALIZATION

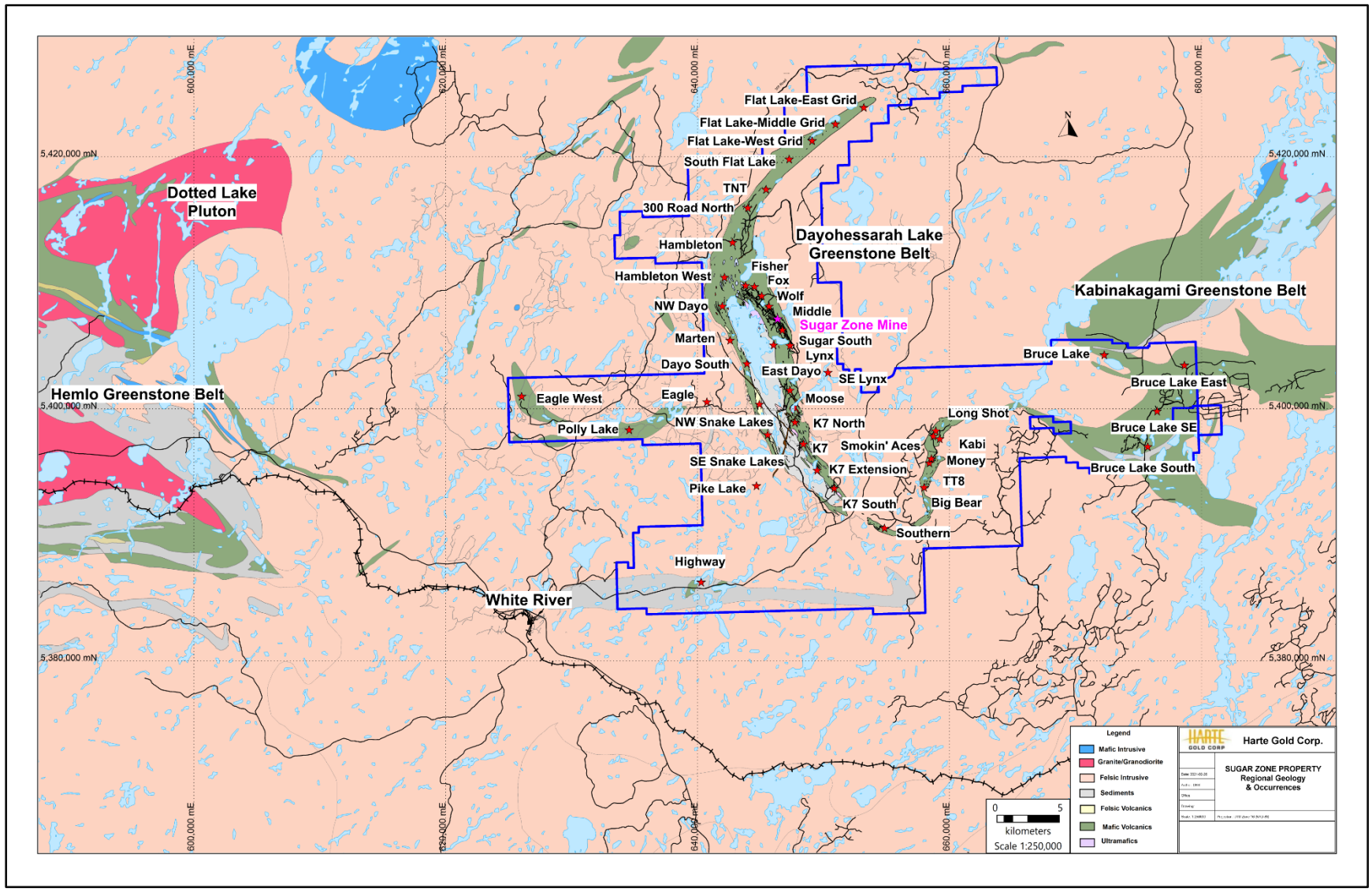
The Dayohessarah Greenstone Belt (the Belt) of late Archean (ca. 2.7 Ga) age is located in the Abitibi–Wawa Subprovince of the Superior Province. The Belt is positioned between two larger greenstone belts; the Hemlo Belt to the west and the Kabinakagami Belt to the east. The Belt trends broadly north and forms a narrow, eastward-concave crescent (Figure 7-1). The Belt is approximately 36 km long and varies from 1.5 km to 5.5 km wide. Principal lithologies in the Belt are moderately to highly deformed metamorphosed volcanics, volcanoclastics, and sediments that have been enclosed and intruded by tonalitic to granodioritic quartz-porphyry plutons.

The Belt is bordered to the east by the Strickland Pluton and to the west by the Black Pic Batholith. The Danny Lake Stock borders the southwestern edge of the Belt. The Strickland Pluton is characterized by a granodioritic composition, quartz phenocrysts, fine-grained titanite, and hematitic fractures. The Black Pic Batholith is similar to the Strickland Pluton, but locally more potassic, with monzogranite phases. The Danny Lake Stock is characterized by hornblende porphyritic quartz monzonite to quartz monzodiorite (Stott, 1999).

The Belt has been metamorphosed to upper greenschist to amphibolite facies. The Strickland Pluton is thought to have been emplaced into the Belt, and imposed a thermal metamorphic event. Most of the mafic volcanics are composed primarily of plagioclase and hornblende. Almandine garnets are widely observed in the clastic metasediments, and, locally, pyrope garnets are present in the mafic metavolcanics (Stott, 1999).

Alteration throughout the Belt consists of diopsidation, albitization, weak magnesium biotization, weak carbonatization, and moderate to strong silicification that accompanied the emplacement of the porphyry dykes/sills and quartz veining. The Belt has been strongly foliated, flattened, and strained. Deformation seen in the supracrustal rocks has been interpreted to be related to the emplacement of the Strickland Pluton. Strongly developed metamorphic mineral lineations in the supracrustal rocks closely compare with the orientations of the quartz phenocryst lineations seen in the Strickland Pluton. This likely reflects a constant strain aureole imposed by the pluton upon the Belt (Stott, 1999). The strain fabric is best observed a few hundred metres from the Strickland Pluton in the Sugar Zone, which has been characterized as the most intensely strained part of the Belt. The Sugar Zone is defined by sets of parallel, mineralized quartz veining; quartz flooding of strongly altered wall rock; thin intermediate porphyry lenses and dykes/sills parallel to stratigraphy and foliation; and gold mineralization.

Stratigraphic layering and top indicators define a synclinal fold in the central portion of the Belt. The synclinal fold has been strongly flattened, and stands upright with the fold hinge open to the south and centred along Dayohessarah Lake.



Source: Harte Gold, 2021.

Figure 7-1: Regional Geology

7.1 Property Geology

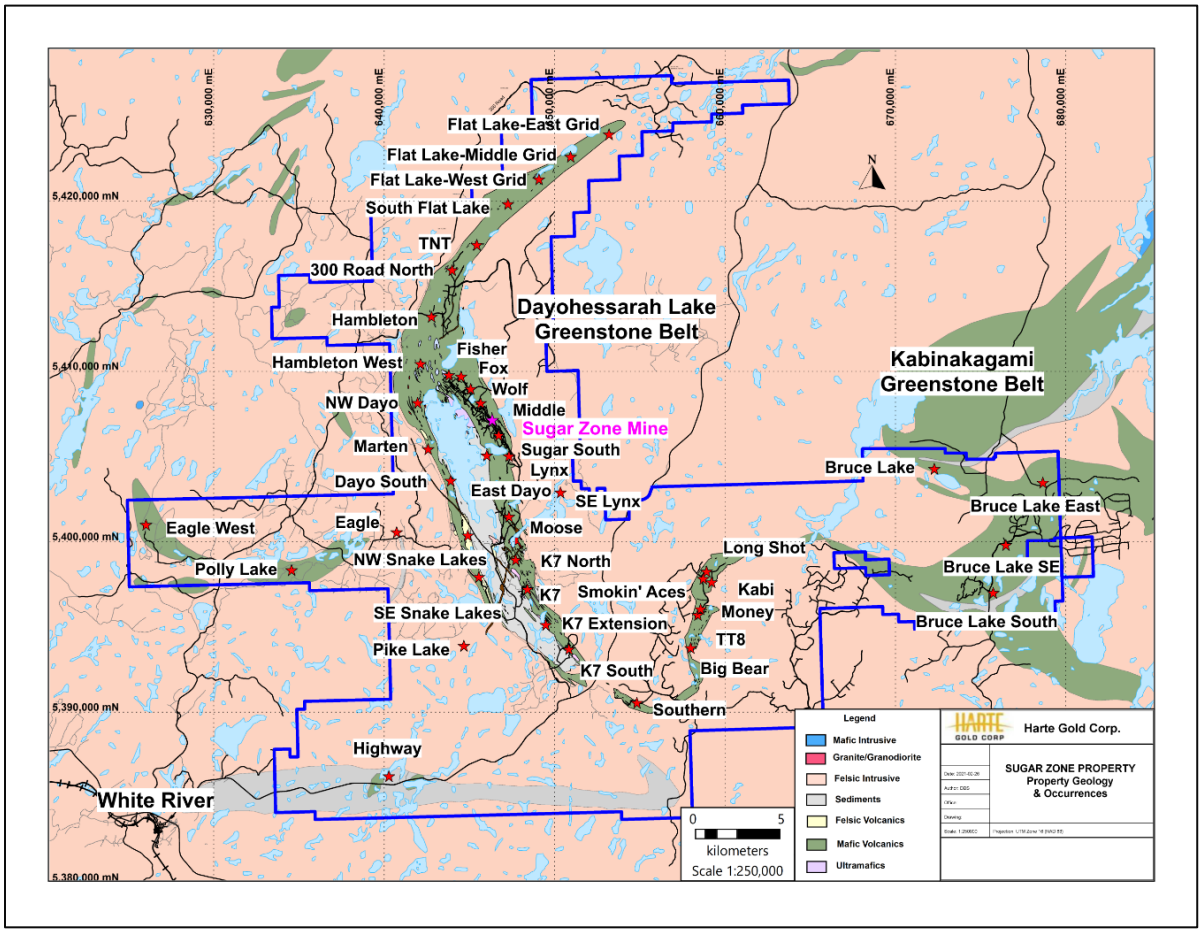
Near Dayohessarah Lake, the Belt is dominated by a basal sequence of massive to pillowed mafic volcanics, commonly with ellipsoidal, bleached alteration pods, overlain by intermediate tuff and lapilli tuff. The tuffaceous units rapidly grade upwards to a sedimentary sequence consisting of greywacke and conglomerates derived from volcanics, sediments, and felsic intrusive sources (Stott, 1999). Several thin, continuous, cherty sulphide-facies iron formations are found in the mafic volcanic sequence. Spinifex-textured komatiitic flows stratigraphically underlie the main sedimentary sequence and can be traced around the north end of Dayohessarah Lake. Also, at the north end of Dayohessarah Lake, mafic and ultramafic sills and stocks underlie the komatiites (Figure 7-2).

Numerous fine- to medium-grained, intermediate feldspar porphyry dykes and sills have intruded the metavolcanic and metasedimentary rocks, and form an intrusive swarm. The intrusions of the intermediate porphyry dykes are most intense east of Dayohessarah Lake. Stott has interpreted the porphyry sills and associated porphyry bodies to be related to the Strickland Pluton. A smaller granitic quartz porphyry body containing some sulphide mineralization is located northwest of Dayohessarah Lake. The porphyritic texture of the dykes/sills is often nearly, or completely, obliterated by the degree of foliation in the Belt, or by the degree of deformation in the Sugar Zone. These intermediate dykes/sills vary in abundance across the Property, but increase in regularity within, and around, the Sugar Zone. There is also a consistent, weak pervasive silicic alteration in the intermediate intrusive rocks, as well as consistently trace amounts of very fine-grained disseminated pyrite.

The major deformation structure recognized on the Property is the Sugar Deformation Zone (SDZ), which trends northwest–southeast for approximately 3.5 km and dips west between −65° and −75°. The SDZ appears to be spatially related to the Strickland Pluton, and is a complex zone with strain intensities varying from strongly deformed, pillow-mafic volcanics to undeformed massive mafic flows, to anastomosing linear zones. Structurally concordant porphyritic intermediate sills form a swarm through the SDZ. Both the mafic volcanics and the intermediate intrusives exhibit moderate linear fabrics along with hydrothermal silicification.

Stretching and foliation of all rock types, except for the later diabase and felsite dykes, increases with proximity to the SDZ. Within and adjacent to the SDZ, basalt flows are foliated and stretched to the point where primary textures become unrecognizable. Widespread ‘mafic agglomerate’ noted in early exploration on the Property has more recently been interpreted as highly stretched pillowed flows. Within, and proximal to the mineralized zones, boudinage of quartz veins and other brittle features is commonly observed.

White to pale grey, fine- to medium-grained, and occasionally pegmatitic felsite dykes intrude the metavolcanic rocks and the mineralization. The dykes are generally narrow, northeast striking, and consist of varying amounts of plagioclase, quartz, and muscovite. The dykes are undeformed, and postdate the mineralization and deformation events.



Source: Harte Gold, 2021.

Figure 7-2: Geology of the Sugar Zone Gold Property

A northerly striking, sub-vertically dipping, dark grey-black, weakly magnetic diabase dyke intrudes the older rocks of the Belt, and cuts across the SDZ. The diabase dyke is aphanitic at the contact and grades into a coarse-grained rock in the dyke centre. The dyke contains coarse-grained, greenish feldspar phenocrysts replaced by quartz-epidote up to 3 cm diameter. A small lateral displacement of the Sugar Zone is interpreted locally on either side of the dyke.

7.2 Deposit Geology

The Upper, Lower, and Footwall mineralized subzones range in thickness from 0.2 m to 6 m, strike at 140° and dip between 65° and 75° to the west. From west to east, the north-northwest striking lithological units hosting the gold mineralization in the Sugar Zone can be sequentially subdivided as follows:

- Hanging Wall volcanics
- Upper Subzone (Sugar Zone mineralization)

- Interzone volcanics
- Lower Subzone (Sugar Zone mineralization)
- Interzone volcanics
- Footwall Subzone (Sugar Zone mineralization)
- Footwall volcanics.

The Hanging Wall, Interzone, and Footwall volcanic horizons consist predominantly of massive and pillowed basalt flows generally striking north-northwest and dipping −64° to the west. Coarse to very coarse-grained, locally gabbroic-textured phases form a significant component of the Hanging Wall mafic volcanic package. These rocks are interpreted as representing portions of massive mafic flows that grade into finer-grained mafic flows, and eventually to pillowed flows. In much of the area in which drilling on the Sugar Zone was carried out, a distinctive, very coarse-grained, mafic volcanic flow was observed consistently about 15 m stratigraphically above the Upper Subzone. Other than this unit, specific mafic flows and intermediate porphyry units are nearly impossible to interpret/distinguish between holes.

The Middle Zone, located along strike between the Sugar and Wolf Zones, shares a similar geological setting, hosted by massive and pillowed basalt flows. Gabbroic-textured flows or sills are as well common. Similar to the Sugar Zone, feldspar porphyry sills often occur adjacent to, or form part of, the mineralized zone. Mineralization styles are similar to those of the Sugar Zone, with gold values commonly occurring in quartz veins and veinlets within an altered zone of mafic volcanic and feldspar porphyry.

The auriferous Wolf Zone lies in the northern extent of the SDZ. Similar to the Sugar Zone, the Wolf Zone is north-northwesterly striking, and westerly dipping; however, there is only one gold mineralized zone. The Wolf Zone is interpreted as dipping −70° to the west and plunging steeply to the south. Wolf Zone-style mineralization is characterized by silicification with fine-grained sulphides and altered porphyry, and is interpreted by Harte Gold to have similarities with Hemlo-style mineralization.

7.3 Mineralization

The mineralized Upper, Lower, and Footwall Subzones of the Sugar Zone lie within the SDZ. They occur within a highly strained assemblage consisting of variously altered mafic volcanic flows, intermediate porphyritic intrusions, and boudinaged auriferous quartz veins. The three zones range in true thickness from about 0.2 to 6 m, and are separated by 20 to 30 m of un-mineralized metavolcanics.

Each Subzone is made up of one or more porphyritic intrusions, flanked by altered basalt, and hosting conformable quartz veins. Alteration within the mafic metavolcanic portions of the subzones consists primarily of silicification that occurs both as pervasive alteration, and as quartz veining, diopsidation, and biotization. The porphyry units of the zones exhibit biotite and silica alteration, but lack diopside alteration.

Gold mineralization mostly occurs in quartz veins, stringers, and quartz-flooded zones predominantly associated with porphyry zones, porphyry contact zones, hydrothermally altered basalts and, rarely, weakly altered or unaltered basalt within the Upper, Lower, and Footwall Subzones.

Fine to coarse-grained specks and blebs of visible gold are common in the Sugar Zone quartz veins, usually occurring within marginal, laminated, or refractured portions of the veins. The visible gold itself is often observed to be concentrated within thin fractures, indicating some degree of remobilization. Quartz veins and floods also contain varying amounts of pyrrhotite, pyrite, chalcopyrite, galena, sphalerite, molybdenite, and arsenopyrite. The presence of galena, sphalerite, and/or arsenopyrite is a strong indicator of the presence of visible gold. Pyrite, chalcopyrite, and rarely molybdenite form a minor component of total sulphides, and do not appear to be directly related to the presence of gold mineralization.

Other mineralized zones have been observed in diamond drilling between, above, and below the Sugar Zone Upper and Footwall Subzones. Most of these intercepts are believed to be quartz veining originating in either the Upper or Footwall Subzone, which have been diverted from the sheared part of the zone, up to 15 m from the main bodies of mineralization. One of these zones is the Lynx Zone, which lies east of the southern end of the Sugar Zone.

The Middle Zone, located between the Sugar and Wolf Zones, may represent the extension of the Sugar Zone to the north. The zone occurs within a highly strained package of massive and pillowed flows exhibiting various degrees of biotite alteration. Gabbro sills and flows are common about the zone. Similar to the Sugar Zone, a weak Upper and Footwall Zone may locally be developed. The zone typically ranges from 0.2 to 6 m in thickness.

Middle Zone gold mineralization is often associated with quartz veins and veinlets hosted within a package of altered mafic volcanic and feldspar porphyry. The gold mineralization is often accompanied by 1%–5% pyrrhotite, and pyrite with local sections of minor galena and sphalerite. Galena is normally indicative of higher grades and the presence of visible gold.

The auriferous Wolf Zone lies along strike of the Sugar Zone, and may represent the northern extension of the SDZ. It is defined as highly strained packages consisting of variously altered mafic volcanic flows and gabbros. The zone ranges in true thickness from 0.2 to 3.0 m.

The Wolf Zone is made up of highly sheared mafic metavolcanics and a network of intrusive, intermediate quartz-feldspar porphyry dykes/sills. Alteration in the mafic volcanic and gabbro units consists mainly of silicification (both pervasive and quartz veining), diopside alteration, and magnesium-rich, brown biotite alteration. Alteration within the intermediate porphyry units consist of mostly silicification, with small amounts of magnesium-rich brown biotite, and no diopside. The zone is observed in trenches to pinch and swell over 30 m.

Wolf Zone gold mineralization mostly occurs in quartz veins, stringers, and quartz-flooded zones predominantly associated with porphyry zones and hydrothermally altered basalts and gabbros. Fine-

grained specks of visible gold are occasionally observed in the Wolf Zone quartz veins. The visible gold itself is often observed to be concentrated within thin fractures, indicating some degree of remobilization. Quartz veins and floods also contain varying amounts of pyrrhotite, pyrite, and occasional galena. The presence of galena is a strong indicator of the presence of visible gold. Pyrite and pyrrhotite form most of the total sulphides, but do not appear to be directly related to the presence of gold mineralization.

8 DEPOSIT TYPES

In the Superior Province, major gold deposits are spatially associated with large-scale regional deformation zones and associated Timiskaming-type metasediments (LeClair et al., 1993). These regional structures are interpreted as zones of transpressive terrain accretion.

Typical greenstone-hosted, mesothermal gold deposits are associated with structurally controlled quartz-carbonate veins hosted by moderately to steeply dipping shear zones displaying brittle to ductile deformational features in low-grade (greenschist) metamorphic rocks. In contrast, the Sugar Zone gold deposit is hosted in medium-metamorphic-grade (amphibolite) rocks that exhibit ductile deformation. The Sugar Zone host rocks have been intruded by felsite and porphyry sills, and exhibit strong association of gold with silica-sulphide-potassic alteration. These characteristics are shared with the Hemlo deposit 68 km to the west.

The Sugar Zone is interpreted as an orogenic, mesothermal gold deposit located in the SDZ, which is an area of high strain. The auriferous zones of the Sugar zone are interpreted to be highly strained altered mafic flows, intermediate to felsic porphyritic intrusions, and boudinaged auriferous quartz veins. Alteration consists primarily of silicification, biotization, and sulphidization. Hydrothermally altered basalt is recognized as a key component of mineralized zones. Commonly in contact with porphyries within mineralized zones, it is strongly silicified biotitic basalt containing significant amounts of pyrrhotite, and pyrite. The Sugar Zone, along with the Middle Zone and Wolf Zone, are three known gold deposits on the SDZ within the Belt.

9 EXPLORATION

This section describes exploration on the Property since 2010. Effective 10 March 2010, Harte Gold became the Operator of the Harte Gold–Corona Joint Venture, and subsequently, on 23 May 2012, Harte Gold made a payment of $2,500,000 to Corona, thus becoming 100% owner of the Property.

9.1 Exploration Work (2010 to 2011)

Information in this section is summarized from Kociumbas and Power-Fardy (2012).

In July and August 2010, Harte Gold contracted Dan Patrie Exploration to cut a grid along the northeastern edge, and around the northern edge of Dayohessarah Lake, totalling 20,475 m. Immediately following completion of the grid, Dan Patrie Exploration conducted ground magnetic and IP surveys on the grid.

During May and June 2011, another grid totalling 60,800 m was completed over the "Fold Nose" area on the northern part of the Belt by JVX Ltd. who also completed ground IP and magnetic surveys over this grid for Harte Gold.

The two geophysical surveys were successful in outlining several linear, chargeable features striking parallel to the regional foliation. The chargeability results, along with the magnetic and conductivity results, were used to create targets for prospecting, as well as exploration drilling (HG holes). A chargeability anomaly up-ice of the Peacock Boulder was thought to be the source of the high-grade gold-bearing boulder, and led to the discovery of the Wolf Zone.

A two-person prospecting program was undertaken between May and October 2011. In all, 180 samples were taken, including four field-inserted gold standards, and four blanks. The prospecting focussed on several targets across the Property, including a Gossan Zone on the western edge of Dayohessarah Lake, a boulder train 500 m east of the northern shore of Dayohessarah Lake, the Lynx Zone, and the IP targets in the Fold Nose area. Overburden around the Fold Nose area was up to 5 m thick, leaving very little outcrop and making prospecting difficult. Gold samples from the boulder train included samples of up to 14.1 g/t Au and 7.3 g/t Au.

A small, five-line grid was completed south of the Gossan Zone in August 2011. The grid totalled 5,200 m running at approximately 60°. The three lines in the middle were all 1,000 m long and spaced at 100 m intervals. The two lines on the north and south ends of the grid were 1,000 m long (north line) and 1,200 m long (south line), and were spaced at 200 m from the other lines. There was no baseline in the grid. Gold samples from the Gossan Zone appeared to have high potential and consisted of large amounts of fine-grained disseminated pyrite and pyrrhotite in a strongly silicified sedimentary unit. Prospecting at the Gossan Zone returned values up to 1.1 g/t Au.

Four mechanical trenches were excavated in 2010 around the discovery of the Wolf Zone. One trench was completed over the Wolf Zone and three others were made over other geophysical targets. Only the trench over the Wolf Zone uncovered significant gold mineralization.

At the request of Harte Gold, between 30 August and 1 September 2011 Geotech Ltd. (Geotech) flew a helicopter-borne geophysical survey over the northern edge of Dayohessarah Lake. The principal geophysical sensors included a versatile time-domain electromagnetic (VTEM plus) system, and a caesium magnetometer. A total of 302-line km of geophysical data were acquired during the survey, covering an area of 26.77 km^2. The survey outlined five potential targets of moderate-to-high conductivity, which are mostly explained by the occurrence of large-scale diabase dykes and sedimentary systems. The survey results appeared to be too large-scale to pick up small, discrete gold-bearing systems.

Between 2 August and 19 August 2011, JVX Ltd. conducted a borehole detection array (pole–dipole) and a borehole directional array (gradient) IP survey on the Property in the immediate area of the Sugar Zone. The surveys were conducted on 11 drill holes. A potential, very large-scale deep gold target was identified below 600 m vertical depth. This target is shown as a large chargeability high.

9.2 Exploration Work (2012 to 2020)

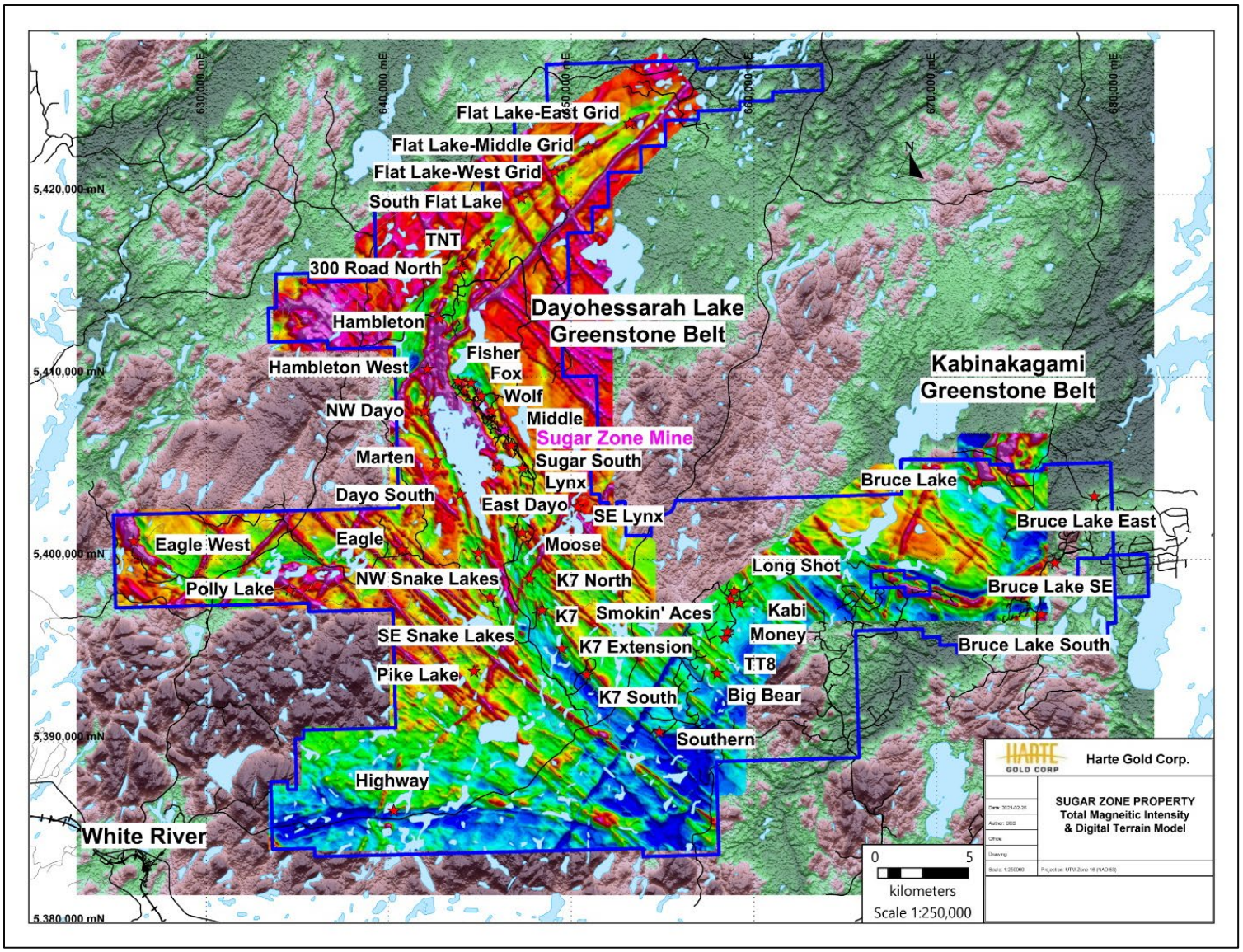
In April 2012, Geotech carried out a helicopter-borne geophysical survey over the Dayohessarah Lake Property, as shown in Figure 9-1 and Figure 9-2. The program was completed as an extension of the airborne VTEM survey conducted in 2011 that totalled 302-line km of data over the northern parts of Dayohessarah Lake and western parts of Hambleton Lake and the shoreline. The 2012 program totalled 1,153 line-km of data and covered the remainder of the Belt.

In an effort to understand the source of the Peacock Boulders, thin sections of three Peacock Boulder samples were sent to Pleason Geoscience for analysis. The boulders returned assay values of 87.30 g/t Au, 52.80 g/t Au, and 37.20 g/t Au. It was noted that the mineralogy and microtextures of the samples were similar to gold-bearing zones at the Hemlo and Musselwhite gold camps.

Between 30 October and 2 November 2012, four mechanical trenches were excavated along the surface exposure of the Sugar Zone. The purpose of the trenches was to expose enough high-grade material from the Lower Zone of the Sugar Zone for a reasonably representative blasting program. The total area of the trenches was 1,799 m^2.

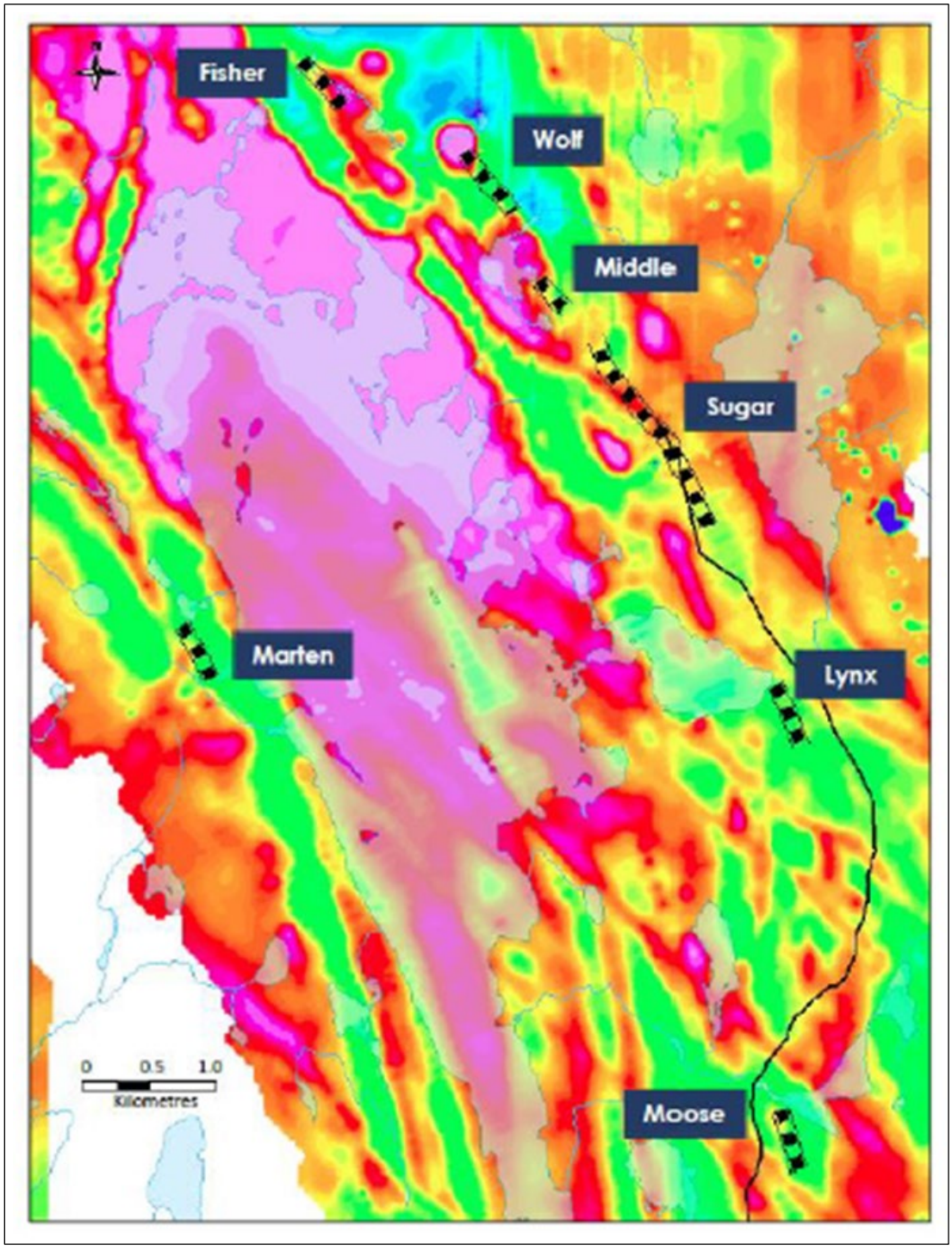
Exploration in the 2013 season included a short prospecting program, where 46 samples were taken and analyzed for gold using fire assay. Two samples returned gold values of 10.2 g/t Au and 0.73 g/t Au. Harte Gold also completed a regional exploration program and IP survey with the objective of finding the source of the high-grade Peacock Boulders, which returned gold values up to 87 g/t Au.

In 2013, Harte Gold continued moving forward with the permitting and optimizing the advanced exploration 70,000 tonne bulk sample at the Sugar Zone. Confirmation drilling in the bulk sample target area returned significant high-grade gold assays, and enabled Harte Gold to redesign the bulk sample target areas to test this high-grade portion of the Sugar Zone deposit.



Source: Harte Gold, 2021.

Figure 9-1: Airborne Electromagnetic/Magnetic Survey Results and Priority Targets



Source: Harte Gold

Figure 9-2: Detail of Airborne Magnetic Map in Sugar Zone Area Showing Relationships of Other Zones

Harte Gold began road construction to provide highway access to the Property and survey work associated with taking certain mining claims to lease. The Company also advanced contract mining and off-site toll processing agreements.

In 2014, Harte Gold continued to advance the Sugar Zone "Advanced Exploration and Bulk Sample Project." Effort focussed on completing the permitting associated with the amended closure plan, completing the road from Highway 631 to the portal site, and overall optimizing the mining plan developed in the 2012 Preliminary Economic Assessment (Buck, Hayden, & LeBlanc, 2012). Harte Gold also completed the lease application process for certain mining claims that cover the Sugar Zone gold deposit and the immediate surrounding area.

Also, in 2014, induced polarization and magnetometer surveys were conducted over a majority of the core mining claims, which generated numerous drill targets. Follow-up ground proofing and drill programs identified the Wolf Zone as the probable source of the high-grade Peacock Boulders, and led to the discovery of the Contact Zone, where a sericite schist was found to have Hemlo-style geochemistry and anomalous gold; in addition, a third mineralized zone known as the Footwall Subzone was discovered 50 m east of the Sugar Zone Deposit.

In 2015, Harte Gold completed the road construction work to link Highway 631 with the Sugar Zone site. Site clearing and initial surface excavation of the portal access to the Sugar Zone was completed by September. The first portal blast was in October 2015, and 155 m of ramp development was completed by year's end.

Harte Gold also undertook a mapping and sampling program in 2015 on the west side of Dayohessarah Lake in an area referred to then as the Gossan Zone (now Marten Zone), and an area west of Hambleton Lake. An altered felsic unit was delineated in the Gossan Zone area, which returned anomalous silver values in the 6–9 g/t Au range with gold values up to 0.57 g/t. Sampling in the Hambleton Lake area returned anomalous zinc values in the 1,940 to 4,970 ppm range. A total of 168 samples was submitted for analysis.

In 2016, Crone Geophysics conducted a 3-D resistivity survey (Escan) over the main grid, extending from the northern limit of the Sugar Zone to the Wolf Zone. Several IP anomalies were identified for future follow-up. During fall 2016 one of the main chargeability trends that Escan identified on trend with a previous gold intersection in drill core was followed up with a small drill program, leading to the discovery of the Middle Zone.

Harte Gold increased its land position in 2017 with the staking of an additional 48,640 ha covering prospective trends associated with the Belt. The acquired ground, along with the Sugar Zone Property, was surveyed with a helicopter-borne triaxial magnetometer and spectrometer survey in April and June 2017. Line spacing varied from 50 to 100 m depending on the block to be surveyed, with a total of 9,797-line km completed.

Superior Exploration completed a very-low frequency (VLF) EM-16 survey during May and June 2017. A total of 33.8 km of surveying was completed over 20 cut grid-lines in the Middle, Wolf, and Fisher

Zone areas. The survey outlined a number of targets that coincide with iron formations, known mineralized trends, and other unexplained responses. A survey over the Gossan (Marten) Zone was also completed at this time, with 4.82 km of surveying over six reconnaissance grid-lines. Additional VLF EM-16 was completed during October and November 2017 in the area of K7. Six reconnaissance lines were run at 200 m spacing. Several weakly anomalous trends were outlined for future ground follow-up or drilling target.

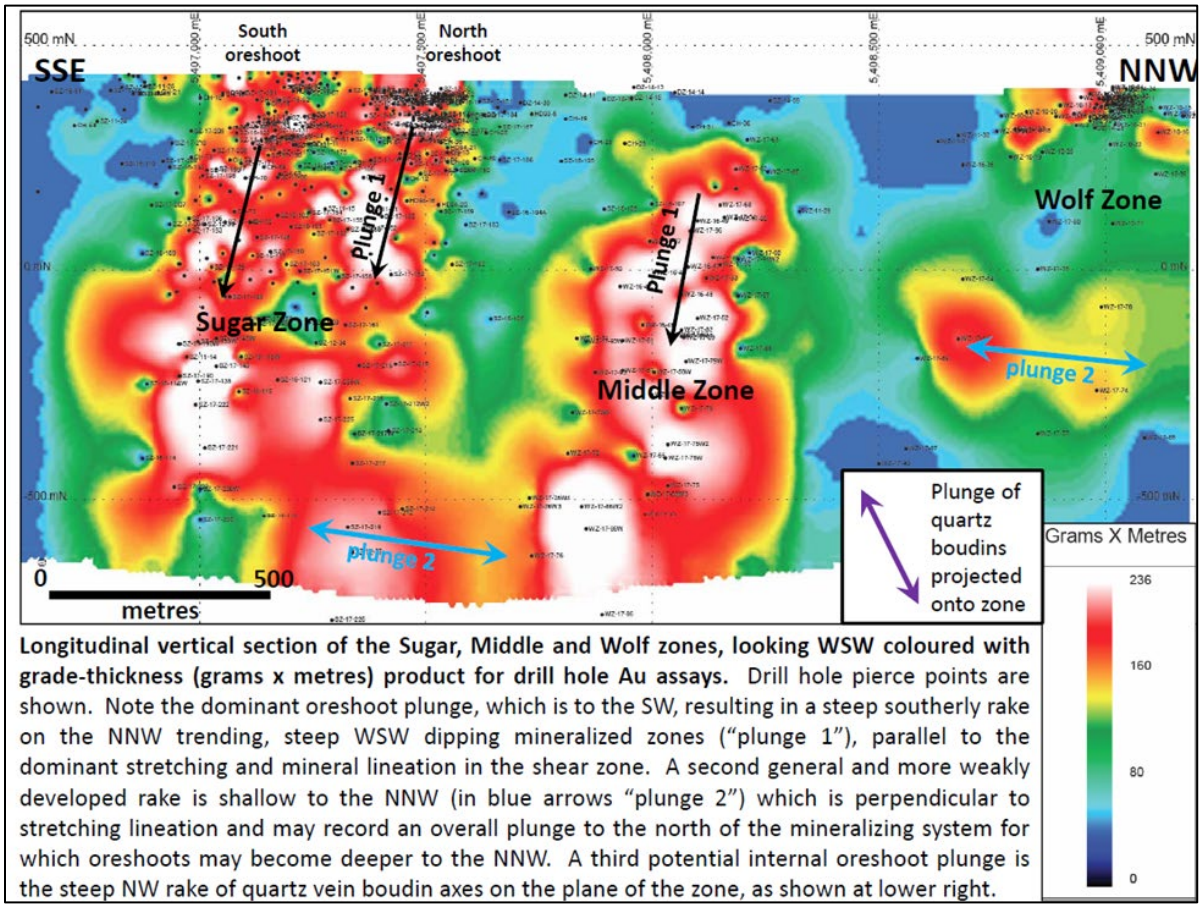
Reaction Geological Consulting Incorporated completed a soil sampling program in 2017 in the Wolf to Fisher and Marten Zone areas. Sampling was done on the B horizon where available using a combination of INAA and multi-acid digest with ICP-OES finish. Samples were analyzed for gold as well as 29 other elements. The Wolf Zone survey consisted of 13 lines spaced at 200 m intervals with samples collected at 25 m intervals. A total of 635 samples were collected in this area. The Marten Zone survey consisted of 15 lines at 200 m spacing with samples collected at 25 m intervals. In all, 467 samples were taken. A number of weak gold and trace-element trends were outlined in both survey areas.

Abitibi Geophysics completed an IP survey in the Lynx Zone area during June and July 2017. A total of 42.4 km of lines were surveyed using a pole–dipole array. A total of 44 chargeable sources were interpreted, with a number recommended for drilling.

During July 2017, David Rys of Panterra Geoservices Inc. undertook a study of the structural setting and controls on mineralization. Surface and underground exposures of the Sugar Zone mineralization and shear were examined, and a suite of 26 samples was collected for petrographic study of the host rocks, alteration, and mineralization mineralogy. Two main plunge directions were identified, a steep SW plunge parallel to the dominant stretching and mineral lineation in the shear zone, and a weaker shallow rake to the NNW perpendicular to the stretching lineation (Figure 9-3). A third potential internal ore shoot plunge is the steep NW rake of quartz vein boudins.

Superior Exploration completed VLF EM-16 surveying between November 2017 and April 2018. Surveys were completed in the Highway and Fisher Zone North areas. A total of 14.9 km of VLF was carried out over 12 reconnaissance grid lines in the Highway Zone area, while 17.1 km of VLF was completed over 11 reconnaissance grid lines in the Fisher Zone North area.

Under the supervision of Blaine Webster, Abitibi Geophysics completed downhole IP surveying on four holes during June and July 2018. An in-hole, pole–dipole, gradient IP survey was carried out in the Middle to Wolf Zone gap area to identify potential chargeability targets for future drilling. Testing of one target between WZ-17-115 and WZ-17-117W was successful in intersecting Middle Zone mineralization in a previously untested area.



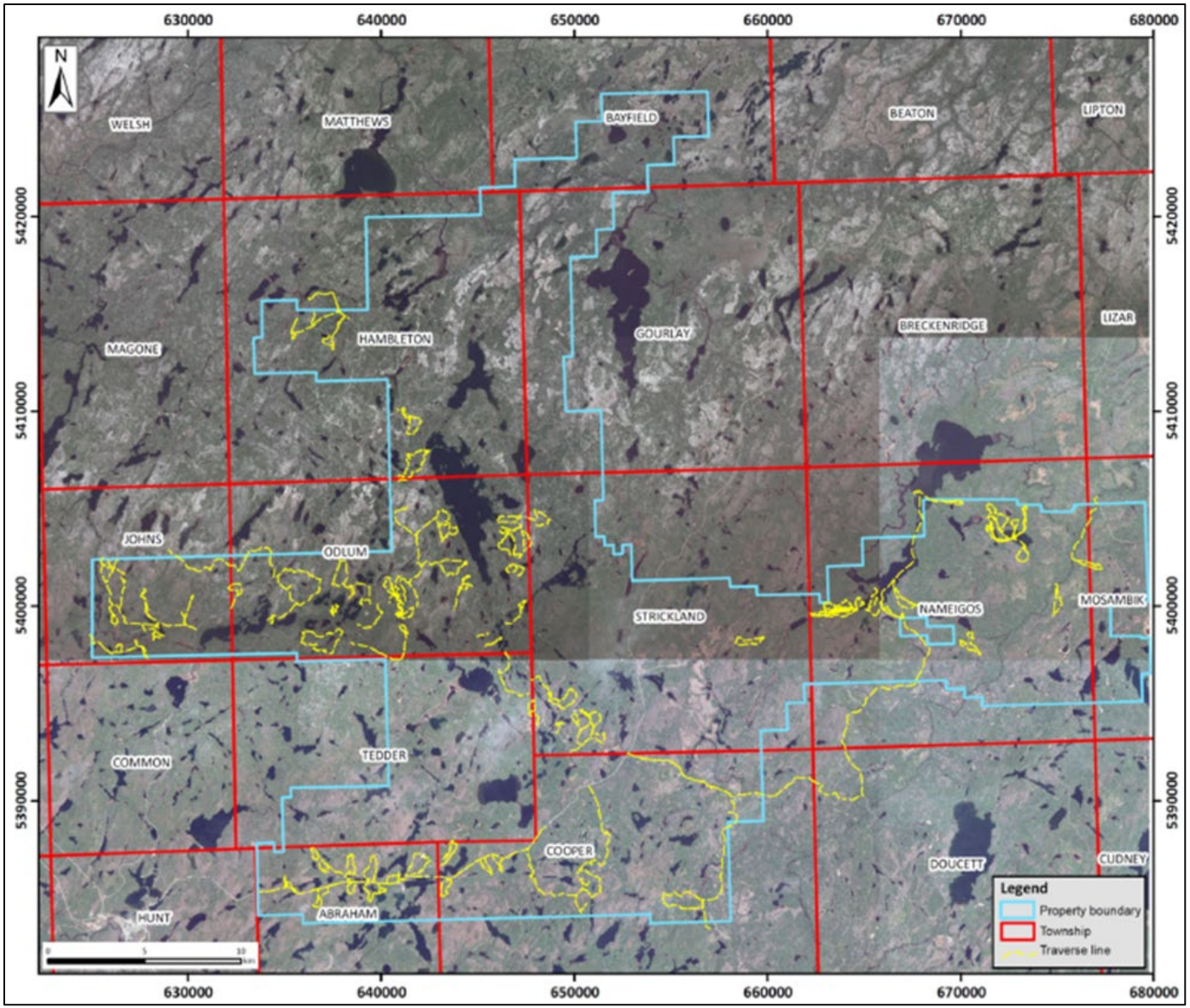
Longitudinal vertical section of the Sugar, Middle and Wolf zones, looking WSW coloured with grade-thickness (grams x metres) product for drill hole Au assays. Drill hole pierce points are shown. Note the dominant oreshoot plunge, which is to the SW, resulting in a steep southerly rake on the NNW trending, steep WSW dipping mineralized zones ("plunge 1"), parallel to the dominant stretching and mineral lineation in the shear zone. A second general and more weakly developed rake is shallow to the NNW (in blue arrows "plunge 2") which is perpendicular to stretching lineation and may record an overall plunge to the north of the mineralizing system for which oreshoots may become deeper to the NNW. A third potential internal oreshoot plunge is the steep NW rake of quartz vein boudin axes on the plane of the zone, as shown at lower right.

Source: David R., 2017.

Figure 9-3: Structural Trends on Contoured Grade Thickness Longitudinal Projection

A combination of Harte personnel and Orix Geoscience 2018 Inc. completed a mapping and prospecting program covering parts of the Sugar Zone Property between June and October 2018 (Figure 9-4). Mapping areas were focussed around Dayohessarah Lake, along Highway 631, the western portion of Nameigos Lake, and the western portion of the Property. A total of 138 rock samples were collected and analyzed as part of the mapping program. Prospecting focussed on the Nameigos, Mountain, and Bruce Lakes, and Kabinakagami River areas in the eastern portion of the Property. In all, 103 rock samples were collected and analyzed. Anomalous gold samples up to 632 ppb Au were obtained in the Bruce Lake area.

Harte Gold, with the assistance of Haveman Brothers Forestry Services Ltd, completed a soil sampling program in the Hambleton Lake area between August and September 2018. A total of 1,365 samples of B Horizon was collected and analyzed for gold and trace elements. Anomalous gold values up to 0.836 g/t Au were obtained.



Source: Harte Gold

Figure 9-4: 2018 Mapping and Prospecting Traverse Lines

During 2019, VLF surveys were conducted at K7 South from 15 January to 17 March, and on the Middle and West grids of the Flat Lake area from 12 March to 23 April. In all, 15.18-line km and 17.3-line km were surveyed at K7 South and the Flat Lake grids, respectively. All lines were spaced 200 m apart. Several weak-to-strong, parallel VLF anomalies were detected on each grid.

From 19 April to 16 May 2019, a four-hole 2,038 m diamond drill program was conducted at the Flat Lake–East Grid to test two, very strong VLF conductors. Although significant widths of Sugar Zone-style alteration were intersected, no significant gold values were encountered.

From 1 April to 26 November 2019 Harte Gold conducted a two-phase prospecting program on the Sugar Zone Property.

In Phase 1 of the prospecting program Maximos Metals Corp. (Maximos) used the data from a February 2019 geophysical study, done by Nanospectra Geophysical Surveying to identify targets of interest for follow-up by Harte Gold prospectors. Maximos interpreted the spectral data from satellite imagery using proprietary software to detect areas with anomalous gold mineralization. Harte prospecting crews evaluated a total of 16 Maximos targets on the ground. The source, or the reason for the anomalous gold response for the majority of the Maximos targets, remains unexplained; however, in areas like Flat Lake and the TNT Zones, altered and mineralized outcrops were discovered associated with the Maximos targets.

Upon completion of the review of the Maximos targets, a Phase 2, more empirical, approach was taken, which generated 15 additional targets for prospecting. The generation of these targets was based on a review of the Harte geological, geophysical, and geochemical (soil, rock) database. This resulted in the discovery of the TT8 Zone, where metallic screen analysis returned grades ranging from 11.1 g/t Au to 247.0 g/t Au from 17 grab samples collected along a 40 m strike length of a 30 cm wide quartz vein.

Between the two phases of prospecting, a total of 392 prospecting person-days were used and 1,393 waypoint stations were recorded, as well as 471 rock samples analyzed.

As follow-up to encouraging prospecting results where gold values of up to 0.253 g/t Au were obtained from mineralized quartz veins in altered mafic volcanics cut by altered feldspar porphyry dykes, a VLF survey was conducted on the TNT grid from 15 October to 16 November 2019. A total of 39-line km of lines spaced at 200 m was completed. Multiple, parallel, weak-to-strong VLF conductors were detected.

From 23 October to 17 November 2019 a seven-hole, 1,929 m drill program was conducted on the TNT grid to test several strong VLF anomalies detected in the earlier VLF survey. No significant gold values were encountered.

From 28 January to 20 February 2020 a 15-hole, 1,639 m drill program was conducted to follow up the high-grade gold values discovered during the 2019 prospecting program at the TT8 Zone. In the TT8 area, drilling confirmed that the TT8 showing consists of a single, coarse-grained, smoky-grey

quartz vein hosting 10% to 15% pyrite, pyrrhotite, magnetite, and at times 1% to 2% chalcopyrite. The vein varies in width from 10 to 30 cm, strikes approximately 30° NE, and dips 20° SE.

Visible gold was noted in ten of the fifteen holes, expanding the mineralization 300 m along strike and 600 m down dip. Highlights include 11.14 g/t Au over 1.18 m, with a higher-grade interval of 33.3 g/t Au over 0.39 m in TT8-20-01, and 33.1 g/t Au over 0.68 m in TT8-20-06.

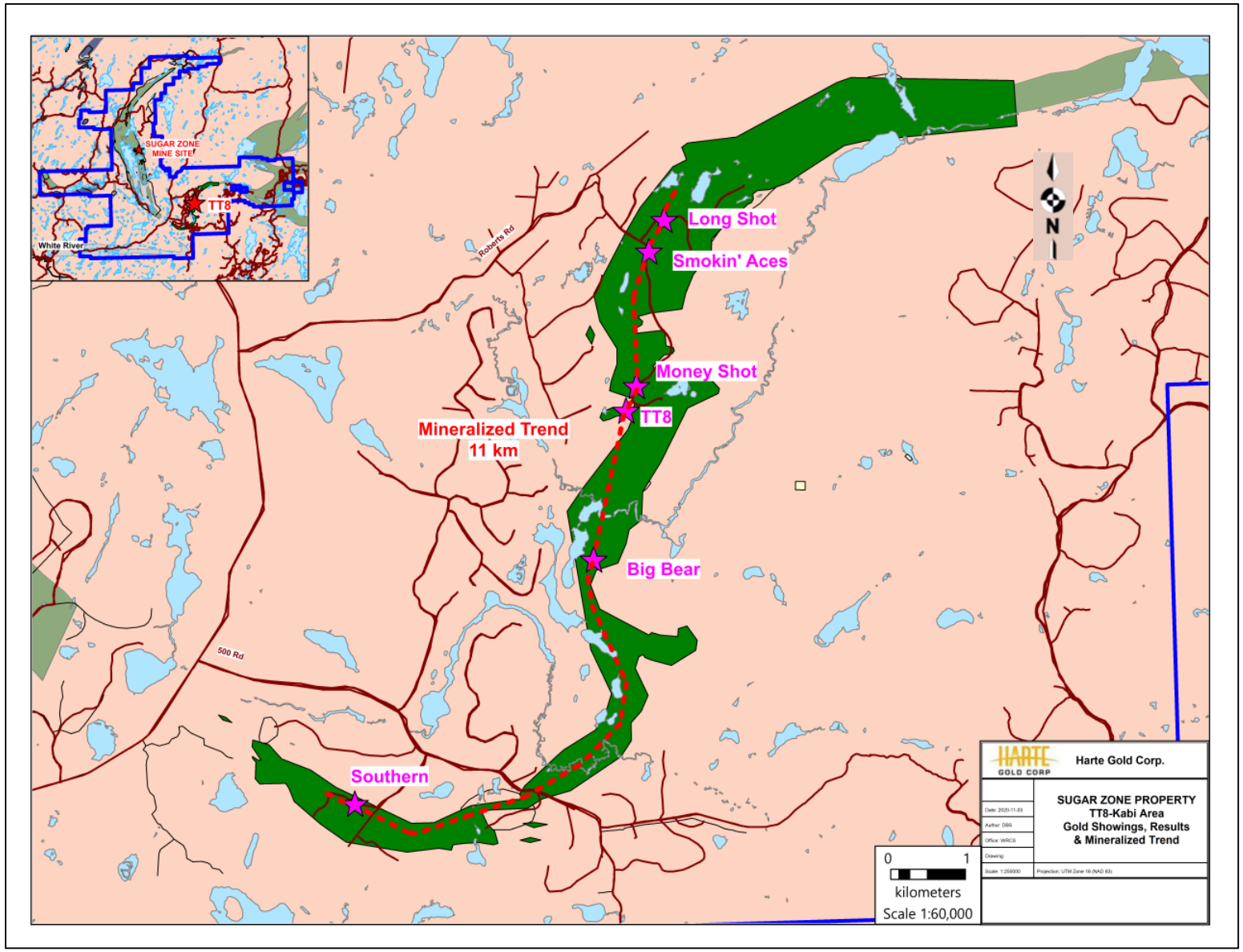
All drill holes intersected the same sequence of rock types, beginning with hanging wall (HW) mafic tuffs, followed by the TT8 quartz vein system, and ending in footwall (FW) greywacke sediments. This package of rocks is intruded by coarse-grained pegmatite and quartz-feldspar porphyry dykes and sills.

From 5 to 7 February 2020 a two-hole, 170 m drill program was conducted to test several weakly to moderately anomalous gold values obtained from the banded iron formation at the Kabi Zone. No significant gold values were intersected.

As follow-up to the encouraging gold values encountered at the TT8 Zone, Abitibi Geophysics Ltd conducted a of 25.95-line km VLF-mag survey over the TT8 grid from 25 to 31 May 2020. Several interesting, parallel, weak-to-strong VLF-mag anomalies were detected and prospected on the ground during the summer of 2020. Prospecting to the north and south of the TT8 Zone resulted in the discovery of 17 km of previously unknown greenstone belt. This prospecting confirms the Kabi greenstone belt to the northeast of the TT8 Zone, and the Belt to the southwest are connected. Furthermore, in addition to the TT8 Zone, five additional high-grade gold showings (Southern, Big Bear, Money showing, Smokin' Aces, and Long Shot) were discovered along an 11 km continuous strike length (Figure 9-5). Gold values range from up to 1.40 g/t Au at the Southern showing to 102 g/t Au at the Money Showing. The gold mineralization is hosted in quartz veins that occur at or near a contact between HW mafic tuffs and FW greywacke sediments, which is the same setting of the gold mineralization in evidence at the TT8 Zone. IP-mag surveys will be conducted over this significant mineralized trend in fall 2020, followed by diamond drill testing.

Soil sampling test surveys were also conducted over the TT8, Smokin' Aces, and Long Shot showings to determine if the local soils were hosting anomalous gold values, or other pathfinder elements that could be used to trace the mineralized horizon along strike. Unfortunately, the results were inconclusive, and no further soil sampling was conducted.

Following Harte's prospecting in the TT8 area, additional prospecting is now being focused on Harte's eastern claims, where Ontario Geological Survey (OGS) geology and airborne geophysics indicates several areas of geologic, structural, and geophysical interest.



Source: Harte Gold, 2020.

Figure 9-5: 2020 Map of TT8 Area Gold Showings

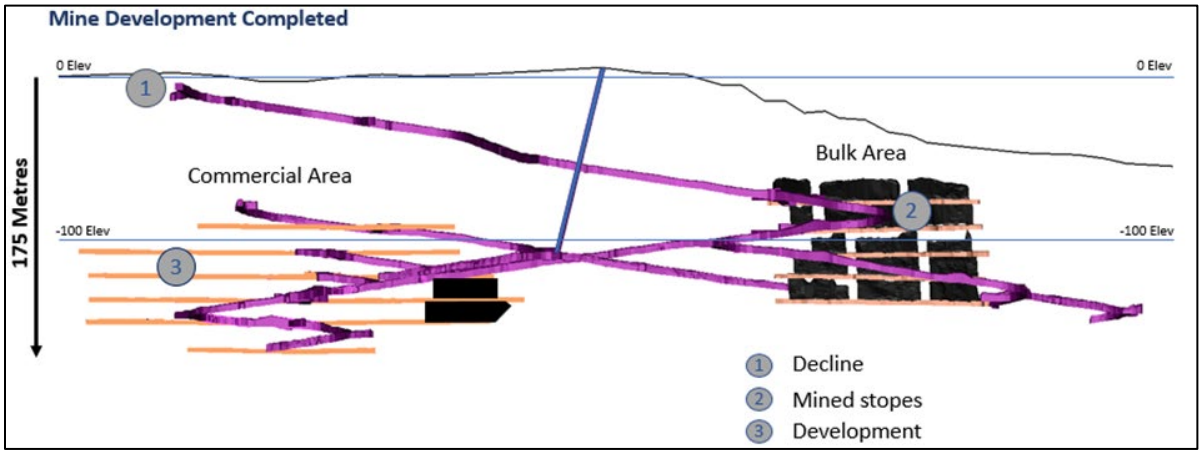
9.3 Advanced Exploration and Bulk Sample

Harte Gold completed a 70,000-tonne bulk sampling program on the Sugar Zone between 2015 and 2017. The program included approximately 960 m of ramp development, and 634 m of horizontal drifting in mineralization on five sublevels at a 15 m vertical spacing, to provide drilling access for longhole mining. A total of 15 stopes was mined from development work completed on the 375, 360, 345, 330, and 315 masl sublevels, as shown in Area 2 in Figure 9-6. An extensive sampling and grade-control program was completed, which included chip, miner muck, crusher muck, and truck muck sampling.

Toll processing of the bulk sample was completed at Barrick's Gold Corporation's Williams process plant in the Hemlo camp. A minimum threshold of at least 5.0 g/t Au for delivered mineralized material was required as part of the toll processing agreement. Shipments of mineralized material to the Williams process plant commenced in April 2016. Shipments continued throughout the year and into March 2017, for a total of 67,424 tonnes at a grade of 8.28 g/t Au. A reconciliation of the bulk sample grades with the Mineral Resource model is presented in Section 14 of this Technical Report.

A Phase 1 Commercial Permit received in January 2017 provided for mining an additional 30,000 tonnes. This permit allowed development of an additional five sublevels on 15 m spacing, as shown in Area 3 in Figure 9-6. This Phase 1 Commercial area was accessed from the existing bulk sample ramp at the −100 m elevation. A total of 561 m of ramp development was completed in 2017, providing access to the 95, 110, 125, 140, and 155 m sublevels. The sublevels noted for the bulk sample were by sea-level elevation, and all levels since the bulk sample are from a datum (zero at the top of the portal). Development on the sublevels totalled 1,055 m, with development material trucked to surface and added to the surface stockpile that is currently being treated at the Harte Gold process plant. Additional development in 2018 amounted to 1,616 m of waste development and 525 m of ore sill development. Production blasting of two stopes amounted to 7,439 tonnes from the 140N and 155N stopes.

Besides the development in mineralization, a 118 m return-air-raise (RAR) was completed to surface. In addition to providing ventilation, the raise has been furnished with an egress ladderway to provide a second means of egress for future mining and development.



Source: Harte Gold

Figure 9-6: Bulk Sample Location and Current Underground Development Longitudinal Projection

10 DRILLING

10.1 Drilling and Channel Sampling

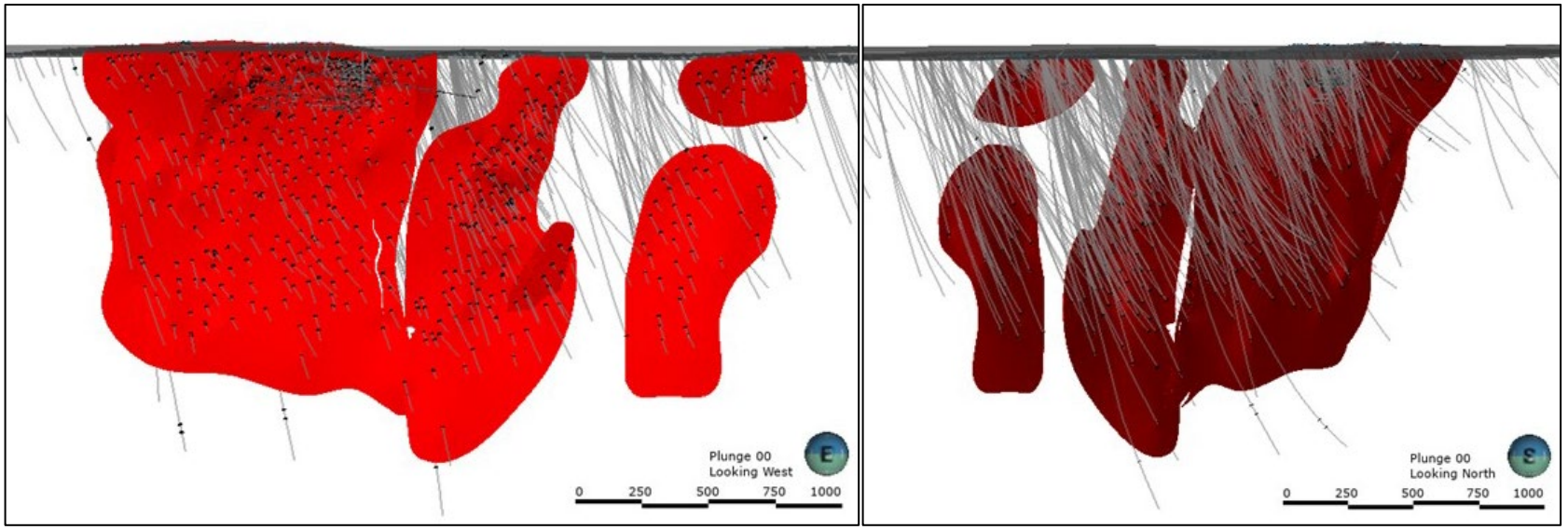
Presented drilling is the accumulation of all drilling along the SDZ. A total of 972 drill holes and 355,593 m of drilling has been completed since 1969. As discussed in Section 6 of this Technical Report, Hemlo Gold Mines initiated the first significant drill program on the Sugar Zone in 1993. The current Mineral Resource estimate is based on 750 drill holes, for a total of 339,234 m that have been drilled since 1993. Table 10-1 summarizes the number of holes by year and operator.

Table 10-1: Sugar Zone Diamond Drill Holes by Year and Operator

Company	Year	No. Holes	Metres
Pre-1993		14	1,197
Noranda	1993	6	800
Noranda	1994	11	1,940
Corona	1998	53	9,922
Corona	2003–2004	38	10,663
Harte Gold	2009	10	2,007
Harte Gold	2010	42	7,115
Harte Gold	2011	31	9,208
Harte Gold	2012	16	7,569
Harte Gold	2013	25	2,798
Harte Gold	2014	52	6,084
Harte Gold	2015	58	5,853
Harte Gold	2016	53	19,957
Harte Gold	2017	279	142,737
Harte Gold	2018	256	112,507
Harte Gold	2019	28	15,236
Total		**972**	**355,593**

Effective 10 March 2010, Harte Gold became the Operator of the Harte Gold-Corona Joint Venture, and subsequently, on 23 May 2012, became 100% owner of the Property.

Since 2017, Harte Gold has collected 2,447 channel samples from underground workings from the 060 Level to 245 Level on both the North and South development fronts, which are subdivided according to the ramp from which they access the ore zone (Figure 10-1).



Long Section Looking West *Long Section Looking North*

Figure 10-1: Longitudinal View of Sugar Deformation Zone Showing Mineralized Wireframes and Drill-Hole Traces

10.2 Harte Gold Drilling Summary

Table 10-2 shows the drilling overseen by Harte Gold since 2009, by Zone and Year.

Table 10-2: Drilling Summary from 2009 to 2019 by Zone and Year

Year	Sugar Zone	Middle Zone	Wolf Zone
2009	Small program to test lateral extents of deposit	-	-
2010	Drilling to extend both the Upper and Lower Zone structures to surface and text continuity down to 600 m true depth	-	Drilling below recently identified Peacock Boulder leads to discovery of Wolf Zone
2011	Resource expansion and testing continuity down to 500 m true depth	-	Infill drilling at Wolf with step-outs towards Sugar Zone
2012	Testing continuity to 1,000 m true depth and step-outs designed to extend strike length	-	-
2013	Infill program targeting potential bulk sample areas	-	-
2014	Infill program of bulk sample area to establish 25 m drill-hole centres	-	Drilling confirmed that Wolf Zone as the source of the Peacock Boulder
2015	Step-outs to the South intersect the Footwall Zones 50 m east of the Upper and Lower Zones	-	-
2016	Resource expansion to 900 m true depth	Testing of a geophysical anomaly between Wolf and Sugar led to discovery of the Middle Zone. Delineation drilling continued through year	-
2017	Infill and step-out drilling to convert Inferred to Indicated categories to 500 m true depth. Increase Inferred category between 500 and 1,000 m true depth. Step-out to the North extends mineralization to within 300 m of Middle Zone	Infill drilling to extend Middle to 700 m true depth with step-outs North towards Wolf Zone and South towards Sugar Zone	-
2018	Infill to increase Inferred category and convert to Indicated category below 500 m true depth. Underground infill drilling from the Lower Zone to increase confidence in Upper and Footwall Zones	Infill drilling to 500 m true depth to convert resources from Inferred to Indicated category. Step-out drilling towards Sugar Zone continued	Infill and step-out drilling to increase resource in Inferred and Indicated categories
2019	Focus on expanding Sugar South Extension adding Inferred resources down to 600 m true depth	Test extents of zone down to 1,000 m true depth	Step-out program from 2018 completed

10.2.1 2020 Underground and Surface Drilling Program

The 2020 drilling program consisted of an underground and surface definition program at the Sugar Zone and a small surface program targeting the Middle and Sugar Zone areas. The underground definition program drilled 10,704 m to define, delineate, and support the 2021 mine plan. The surface program drilled 2,725 m to support the 2021 Middle Zone mine plan, and 357 m to expand Inferred Resources in the South Extension of the Sugar Zone.

10.2.2 Drill Contractors

Chibougamau Diamond Drilling undertook Corona–Harte drilling in 2009. Drilling was on the Sugar Zone proper.

More Core Diamond Drilling, Blackhawk Diamond Drilling, Ed Core Diamond Drilling Services, Major Drilling Ltd., Chibougamau Diamond Drilling, and Foraco Canada Ltd. provided drill rigs from 2009 to 2019. Currently Chibougamau and Foraco Canada are the active contractors.

10.3 Drilling/Channel Methods

10.3.1 Core Drilling

Surface drill size across the Operation is NQ core (47.6 mm diameter). Underground drill size has been AQTK (35.5 mm diameter) due to the use of smaller air/hydraulic drills.

A lack of surface overburden around the Project area decreases the potential for issues at start of hole, resulting in fewer restarts because of deviation during the casing process. Drill holes can “lift” and “drift” due to rock conditions, which can be exaggerated at depth.

Transportation of the core from the logging facility to the core laydown is the geologist’s responsibility.

10.3.2 Channel Sampling

Channel samples are collected from active underground headings using a hammer or hammer and chisel. Samples are collected at intervals that are lithology dependent, and placed in individual sample bags. Tags are placed in each bag, and the location, round number, and sample intervals are recorded in the sample booklet.

10.4 Harte Gold Geological Logging

A set geological legend for the Property areas is used for surface, underground, channel, and drill core. The lithologic types covered included surface sedimentary, igneous (altered and unaltered equivalents), volcanic (altered and unaltered equivalents), and tectonic lithologies. Sampling is bounded by geological considerations with a minimum sampling width of 0.3 m and a maximum interval of 1 m.

Harte Gold is currently reviewing logging definitions and standards to incorporate all relevant geological data that will further improve the geological understanding of the deposit.

10.4.1 Drill Core

Logging of drill core is done at the site core facility. Core is unboxed and inspected, and geotechnical work is done prior to logging. It includes:

- Confirming run lengths
- Identifying natural and mechanical breaks
- Assigning rock quality designation and recovery
- Ensuring that boxes and blocks are free of errors.

Recovery is recorded for quality assurance and geotechnical purposes. Any lost or ground core is also noted within the log.

The basic geological log includes:

- Primary lithology
- Alteration
- Mineralization
- Structures and foliation
- Veining
- Texture
- Samples.

At the Operation, core is rolled to align foliation with the top of the box. This enables cut lines to be quickly drawn when samples are required. Core is photographed both wet and dry (Figure 10-2) prior to being sawed, sampled, and bagged.



Figure 10-2: Example of Wet/Dry Core Record

10.4.2 Channel Sampling

Channel sampling is conducted at active headings in the underground. All faces are thoroughly washed and prepped before a geologist begins work.

A basic channel log contains:

- Primary lithology
- Alteration
- Mineralization
- Structures and foliation
- Veining
- Samples.

At the Operation, a geologist records the channel information from every round. A 1.2 m channel line is marked up from the floor. As shown in Figure 10-3, samples are marked up along this line, honouring lithologies and/or alteration. Each round is photographed with the sample intervals, round number, grade call, and advance instructions.



Figure 10-3: Example of a Channel Record

10.5 Collar and Downhole Surveys

10.5.1 Drill Holes

10.5.1.1 Surface Collar Survey

A two-part process for collar surveys is employed around the Property. The drill collar is recorded using a Trimble 7x handheld GPS unit with an accuracy of ±40 cm. The casing orientation is recorded using a Devico north-seeking DeviAligner.

10.5.1.2 Underground Collar Survey

Harte Gold's underground surveyors mark up the initial collar location. From this starting location a Devico north-seeking DeviAligner is used to align the mast for each hole. The DeviAligner data are recorded for each hole. The completed hole is identified using a rotary drill and tag identifying the specific hole. These holes are then recorded by Harte Gold's underground surveyors using a 2 ft BQ rod that is used to standardize the collar survey from hole to hole. The DeviAligner is also used to confirm drill-hole orientation at the collar if there is uncertainty from the initial rig alignment.

10.5.1.3 Downhole Survey

Downhole surveys are taken using a REFLEX EZ-TRACK downhole survey instrument, with measurements taken at 30 to 50 m down hole. Though very few holes have been stopped due to deviation, the holes are constantly monitored within the first 100 m.

10.5.2 Channels

Channel collars are recorded as a length from the nearest survey control point. This is accomplished using a Leica D2 distometer. Multiple measurements are taken for each round to confirm positioning of the round from the control point. Control points are advanced regularly with development. A round is typically never 25 m beyond the nearest survey control point.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Historical data (i.e., pre-1993) for the Property have not been reviewed, and none of the pre-1993 holes are used in the current Mineral Resource estimate.

11.1 Sample Preparation and Analysis

11.1.1 Diamond Drill Core Samples

When a geologist identifies a zone of interest for sampling, a technician checks that the sample intervals correspond to the recorded depths, and that the sample identification numbers for each interval are correct before proceeding to collect the sample. This process involves checking the digital log, the completed sample stubs, and the intervals identified on the core barrel.

Core trays returning from the drill rig are placed on stands in ascending order of hole ID and box number for the technicians and geologists to take into the logging facilities. In case of overflow, the trays are placed in the designated area in ascending order of hole ID and box number until the geologist has finished with the previous hole and the trays can be moved for processing. Prior to 2017, all drill core was transported to the logging facility in White River for processing, as no facility existed at site.

A saw line is drawn on the core as a solid red line and positioned to provide symmetry to the sample. When the core is cut, it is split such that the two halves have symmetrical structures and/or mineralization.

Core is sampled at the width of the vein with a minimum sample length of 30 cm with 1 m shoulders on either side. Lithologic boundaries are honoured throughout the process.

11.1.2 Channel Samples

Channel samples are recovered from every underground face during sill development, with few exceptions. Samples are collected in such a manner that the sample represents the interval in both geologic characteristics (e.g., mineralization, structure, lithology) and weight. Samples are sealed at the face and delivered to the Harte laboratory for analysis. See Section 10.4.2 and Section 10.5.2 for further description of Channel sampling practices.

11.1.3 Core Storage and Security

All boxes of drill core from within the Sugar Zone drilled during the 1993–1994, 1998, and 2004 diamond drill programs are stored in the drill-core yard behind the coreshack in White River, Ontario. The rest of the drill core drilled during the 1993–1994, 1998, and 2004 diamond drill programs is stored on pallets to the side of Road 305. All drill core from the 2009, 2010, and 2011 diamond drill

programs is stored in the drill-core yard behind the coreshack in White River. Since the start of the 2017 drilling campaign, all core has been stored at site.

11.1.4 Analytical and Test Laboratories

To date, the analytical laboratories used for drill-core analyses are independent commercial laboratories. Prior to 2009, the primary analytical laboratory was Accurasay in Thunder Bay, Ontario. Since 2009, Harte has used Activation Laboratories (Actlabs) in Thunder Bay, Ontario.

Channel samples have been processed at the Harte Gold site laboratory since 2019. Channel samples during the bulk test phase were processed at the Wesdome laboratory in Wawa, Ontario between 2017 and 2019. Pulps from the Harte Gold laboratory are delivered to Actlabs every month as part of an ongoing umpire program.

In 2010 the Standards Council of Canada (SCC) assessed Actlabs to achieve an ISO 17025 certification, which it has maintained since that time.

Table 11-1: Summary of the Involvement of Each Laboratory by Year and Analysis Type[1]

Laboratory	Sample Type	Location	Year	Description	Over Limit
Accurasay	Drill core	Thunder Bay, Ontario	1933–2009	Fire assay; 30 g; AAS Finish	>3 g/t pulp metallic
Actlabs	Drill core	Thunder Bay, Ontario	2009–2019	Fire assay, 30 g; AAS Finish	> 3g/t gravimetric;
Wesdome	Channel	Wawa, Ontario	2016–2019	Fire assay; 30 g; Gravimetric Finish	>10 g/t pulp metallic
Harte Gold	Channel	White River, Ontario	2019–2020	Fire assay; 30 g; Gravimetric Finish	

Note: [1] Where this is known.

Figure 11-1 summarizes the typical sample preparation and analysis flowsheet the Harte Gold site laboratory and Actlabs use for gold assays.

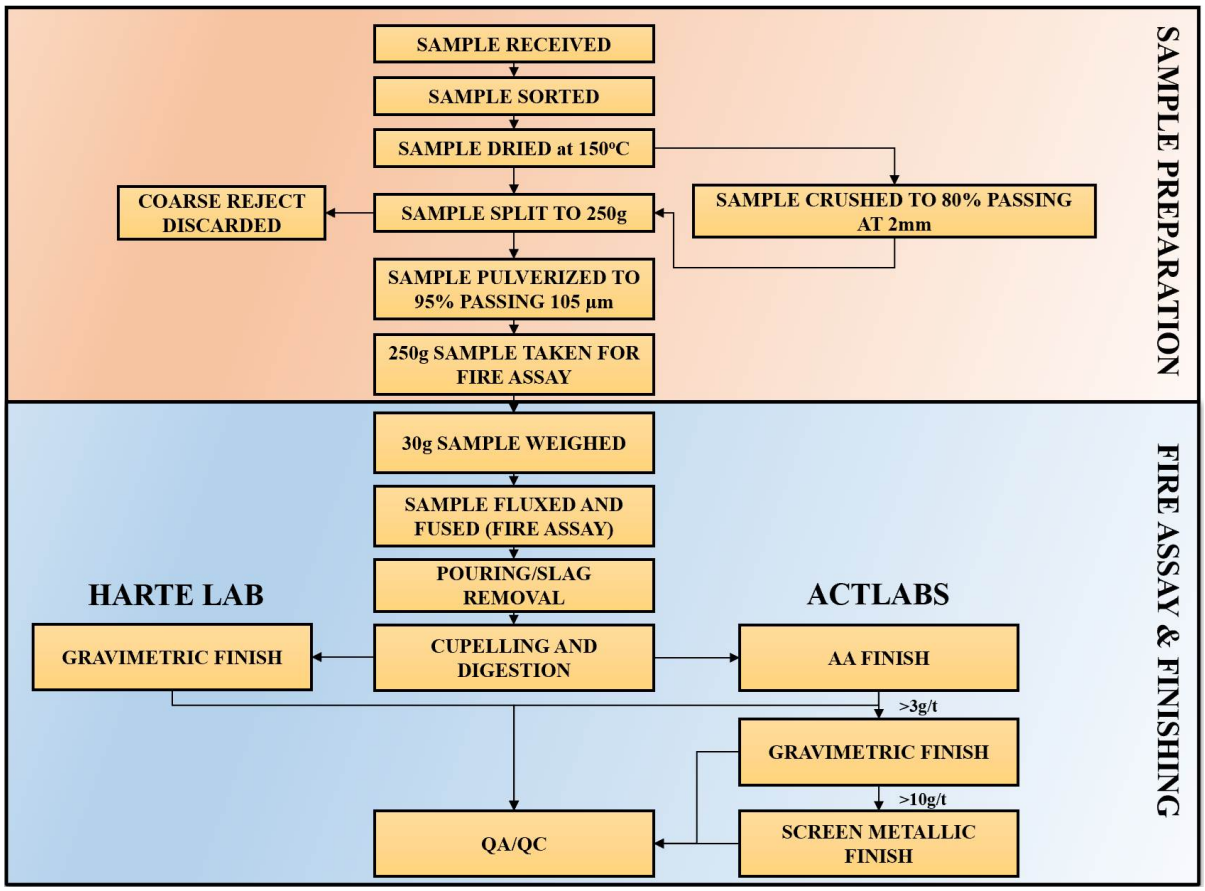


Figure 11-1: Sample Preparation and Analysis Flowsheet

11.2 Quality Assurance and Quality Control Programmes

Certified reference materials (CRM or Standards) and blanks are inserted into the drill core sample stream at regular intervals to monitor laboratory accuracy and precision, and sampling sequencing and precision. Similarly, CRM, and as of 2020, blanks, duplicates, and replicates are inserted into the channel sample stream at regular intervals. Figure 11-2 details the generic sample QA and preparation process flow.

11.2.1 Standards (Certified Reference Materials)

Three standards are currently in circulation at the Operation, and cover material from low- to mid-range grades. Nine different standards have been employed on the Operation since 2009. OREAS has supplied all but one of these standards; the exception was a CDN Resource Laboratories CRM. Drill-core standards have an insertion rate of 1 in 20 samples. In July 2020, the channel sample insertion rate was increased from 1 in 25 samples to 1 in 20, to increase CRM frequency rates.

11.2.2 Blanks

Harte Gold employees collect blank material from a location known to be devoid of mineralization, a pluton 10 km north of White River.

New blank material has been sourced from an outcrop of mafic volcanic material located 2 km from the mine site along the access road. Round-robin testwork was completed by 1 October 2020. Actlabs Thunder Bay, AGAT Laboratories (AGAT) Thunder Bay, SGS Cochrane, and the Harte site lab processed 30 samples each, for a total of 120 samples. Nearly all submitted samples returned values below or at the detection limit (Table 11-2), well below the established failure threshold of 10 times the detection limit.

Table 11-2: Summary of Round-Robin Testwork for Site-Sourced Blank Material

Laboratory	Method	Samples	Detection Limit	Failure Limit	Mean
AGAT	Fire Assay AA	30	0.002	0.02	0.002
Actlabs	Fire Assay AA	30	0.005	0.05	0.004
SGS	Fire Assay AA	30	0.005	0.05	0.003
Harte	Fire Assay Gravimetric	30	0.03	0.30	0.031

11.2.2.1 Drill Core

Coarse blanks are submitted with drill core at a frequency of 1 in 20 samples.

11.2.2.2 Channel Samples

As of July 2020, coarse blanks are inserted into the channel sample stream at a rate of 1 in 20 as a “grab” sample. Prior to this, blanks were last inserted into the sample stream at the initiation of the bulk sample project in 2016.

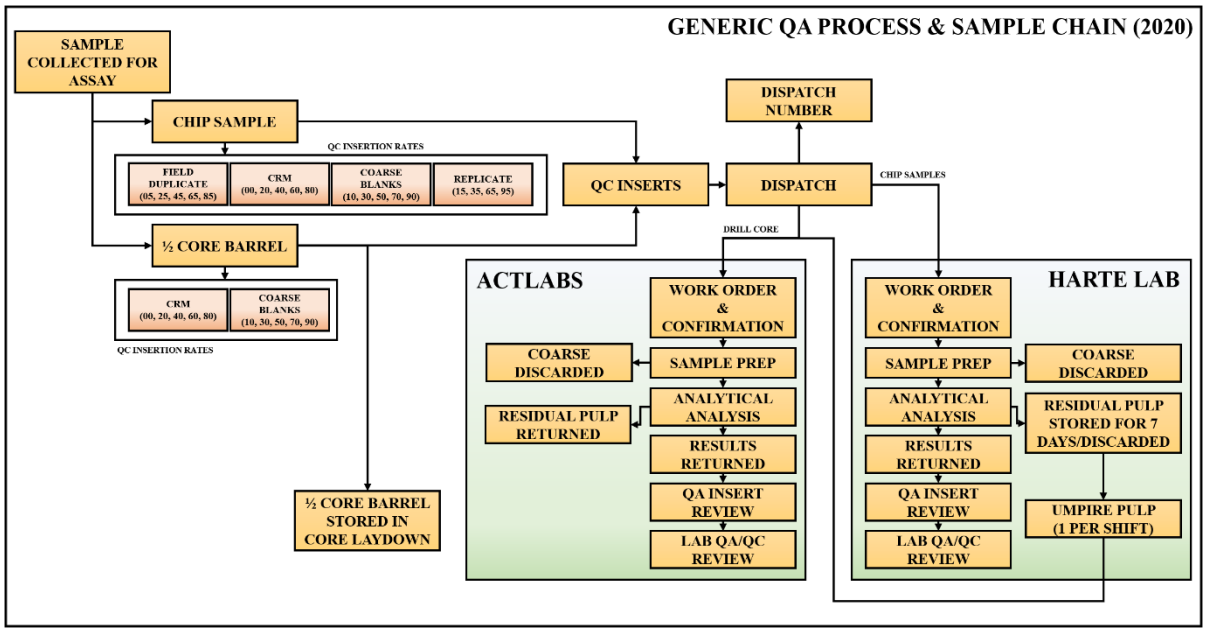


Figure 11-2: Showing a Generic Sample Preparation Flowsheet for Current Processes

11.2.3 Duplicate Samples

11.2.3.1 Drill-Core Samples

Currently only pulp duplicates (i.e., second fire of pulp split) and replicates (i.e., coarse splits) are run for drill core at Actlabs's discretion. Pulp duplicates are run at a frequency of 1 every 10 samples, and replicates are run at a frequency of 1 every 60 samples.

As of 1 December 2020, Harte delivered 200 pulp rejects to SGS Cochrane for secondary umpire check. This package included several samples from the Middle and Sugar Zones from the 2017 drill program through to the 2019 drill program. Moving forward, Harte will deliver a minimum of 5% of the assay pulps to a secondary lab for umpire checks.

11.2.3.2 Channel Samples

Since July 2020, Harte systematically runs four types of duplicates to assess the precision of sampling method and assay analyses:

1. Field duplicates
2. Pulp duplicates (i.e., second fire of primary pulp split)
3. Replicates (i.e., coarse split)
4. External umpire analyses (i.e., secondary lab analysis of pulp reject).

Prior to July 2020, only pulp duplicates were run at the discretion of the Harte lab.

Field duplicates are collected at a frequency of 5% to 10% or 1 every 10 to 20 samples. The duplicate sample is collected at the active face. As this initiative began in January of 2020, full-face duplicates are being collected from two levels of development to build a baseline database.

Pulp duplicates are reported at a frequency of 1 every 10 samples from the Harte site lab. Replicate samples are reported at a rate of 1 every 20 samples as indicated by the Harte Gold geology team.

Umpire assays are delivered to Actlabs monthly. To date, umpire assays are available for 3.6% of the channel assays contributing to the current Mineral Resource estimate update. Umpire assaying procedures have been modified to increase the umpire analysis of channel samples to target 5% or better of the available monthly channel samples.

11.3 Density Analysis

Actlabs previously undertook drill-core density analyses by using a combination of water immersion and gas picometer methods. Following completion of the bulk sample in 2017, the database was updated with 179 bulk density measurements, with values varying between 2.53 t/m^3 and 3.02 t/m^3, with a weighted average of 2.73 t/m^3.

Beginning in December 2020, drill core will be analyzed on site using the water immersion method as a standard procedure of measuring rock with low porosity. This will allow determining density by lithology and mineralized zone. Periodic umpire samples will be sent to a third-party laboratory for confirmation of results.

11.4 Sample Security

11.4.1 Drill-Core Samples

Sample security measures include moving core from the drill site to the Harte Gold core facility at the end of each shift, and tracking sample shipments via chain-of-custody forms.

Core is stored in a laydown by the site security check-point.

Harte is of the opinion that core storage is secure because the site is remote, access is controlled by Harte, and a representative is always on site.

11.4.2 Channel Samples

Every shift the shift Geologist collects channel samples and delivers them to the site lab for analysis. The Harte Gold Laboratory records sample numbers as samples are entered into the system. The sample numbers stay with the sample throughout the entire process to minimize the potential for mislabelling.

11.5 Sample Storage

Pulp rejects are returned from the assay lab to Harte Gold's warehouse in the town of White River. Pulps are stored in dedicated sea cans, in bar-coded kraft paper pulp bags within cardboard pulp trays, boxes, or buckets.

Channel pulps are held for a week following analysis. Pulp rejects selected for umpire checks are delivered to Actlabs, and the remainder are discarded. Starting in 2021, channel sample pulp rejects will be stored for a minimum of six months to a maximum of one year.

11.6 Geological Database

The Harte Gold site geology team in White River, Ontario, manages the drill-hole and channel database; in April 2020 the team audited the database for accuracy, overlapping intervals, missing data, and so on.

Figure 11-3 outlines the current data flowsheet.

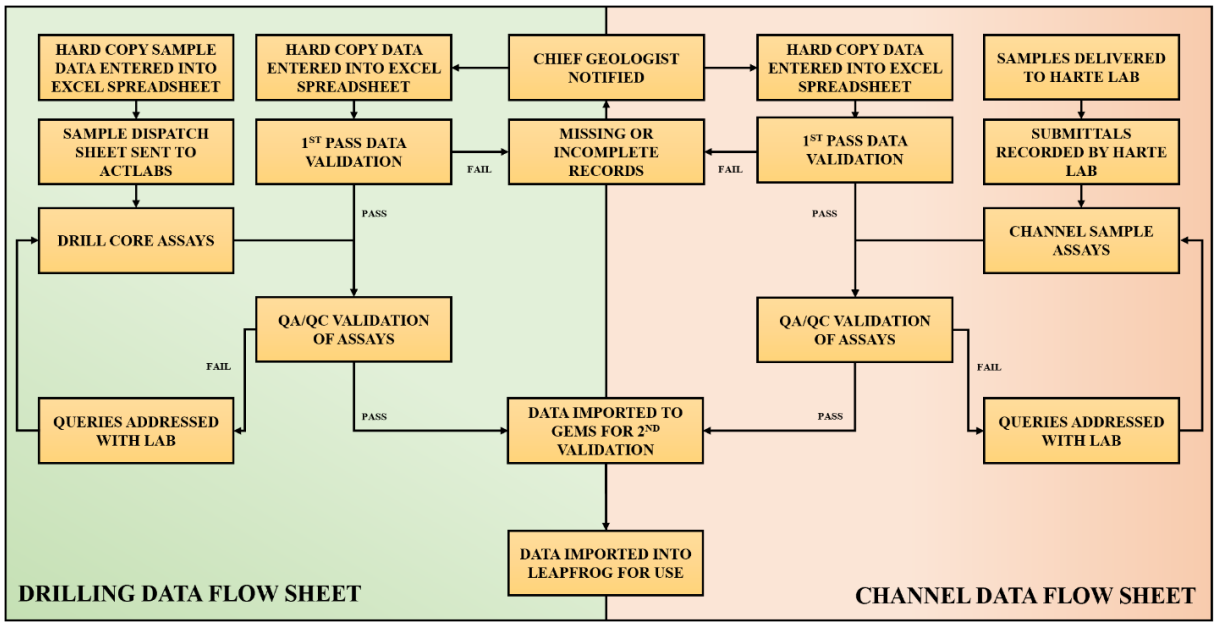


Figure 11-3: Sample Data Flowsheet

Drill-hole data (i.e., logs) are captured on Excel templates with extensive pick-lists, constraints, and validation rules. For a first-stage validation the Harte site geology team reviews completed logs prior to being imported into GEOVIA GEMS. This stage checks that drill-hole collars are plotting in the correct locations, and that the loaded drill-hole data are complete. Completed logs are stored digitally on the site server.

All channel data are captured on paper (i.e., face sheets). Harte geologists enter the data into a series of Excel templates with extensive pick-lists, constraints, and validation rules. The Harte site geology team regularly checks the digital files are against the paper copies. Data are imported into GEMS for a first-stage validation. This stage checks that channel collars are plotting in the correct locations, and that loaded channel data are complete. Face sheets are held on site, and a copy is stored digitally.

Assay data are received from Actlabs and Harte's internal laboratory. Data are delivered directly to the Harte geology team and imported into Excel templates containing extensive input and quality assurance/quality control (QA/QC) constraints. Any validation errors are reported back to the reporting laboratory following Harte Gold's QA/QC procedures. A second-stage validation occurs using GEMS to confirm that imported assay data are assigned to the correct drill-hole or channel sample; no assay overlaps exist; and a final review of the certificate file versus import is conducted. All assay certificate files are stored digitally on the site server.

Once the data are deemed clean, they are imported into Leapfrog software for further analysis.

It is the QP's opinion that sample preparation, security, and analytical procedures for the Property drill program were adequate for the purposes of this Mineral Resource estimate.

12 DATA VERIFICATION

12.1 Database Verification

Data verification undertaken by the QP has shown no significant issues with the integrity of historical data. The sampling techniques and data are considered to be of sufficient quality to carry out Mineral Resource estimations for the Sugar Zone Mine.

Visual validation of the drill-hole locations and mineralized intersections was undertaken and verified against hard copy drill sections. Relative to each other and the cross-sections provided, the drill holes used as the basis for the Sugar Zone Mineral Resource estimates were considered acceptable for classification and reporting under NI 43-101 guidelines.

The QP has verified the data which underpins the Mineral Resource estimate contained in this Technical Report. The QP is of the opinion that the data verification procedures adequately support the integrity of the data used in the Mineral Resource estimate.

12.1.1 Harte Gold Data Checks

12.1.1.1 Drill-Hole Data Entry

Drill-hole data are collected in Excel spreadsheets with multiple pick-lists and validation checks. For a first-stage validation these data are put into a template for import into GEMS. This stage checks that the drill-hole collar is plotting in the expected location with the correct geometry, and that the geologic data are complete. A second-stage validation occurs when assay data are returned from Actlabs. This includes QA/QC checks and a final check in GEMS to confirm no sample overlaps and proper orientation of assays. Ultimately, the data are imported in Leapfrog Geo and checked again.

12.1.1.2 Channel Data-Entry

All underground channel data are initially captured on paper, then transferred to a series of Excel templates with extensive pick-lists and data validation rules. Channel data are imported into GEMS after assays have been returned and QA/QC checks have occurred. This stage checks that the channel collars are plotting in the expected location with the correct geometry, and that the data are complete. Ultimately, the data are imported into Leapfrog Geo and checked again.

12.1.1.3 Quality Assurance/Quality Control

QA/QC data are reviewed on a continuous basis as data are imported into the datasets. Since the beginning of 2020, comprehensive QA/QC reports are generated and reviewed on a monthly basis. Actions prompted by the review are implemented, and again reviewed the following month.

QA/QC data reviews include:

- Control plots: assay standard values are compared against expected values for each standard or blank.
- Correlation plots: original assay data are compared to repeat assay data; original assay data are compared to umpire assay data.
- Half absolute relative difference (HARD) plots.

Drill-core QA/QC data are held to the following validation rules:

- CRM results greater than 3 standard deviations from the expected mean constitute a failure.
- CRM results greater than 2 standard deviations from the expected mean constitute a warning.
- Blank results greater than ten times the laboratory detection limit constitute a failure.
- Duplicate samples with greater than 10% variance are investigated and re-assayed if necessary.

Due to space constraints in the drying and fire assay ovens at the Harte Laboratory, the channel sample QA/QC program is modified slightly from the drill-core QA/QC program.

Channel sample QA/QC data are held to the following validation rules:

- CRM results greater than 3 standard deviations from the expected mean constitute a failure.
- CRM results greater than 2 standard deviations from the expected mean constitute a warning.
- Blank results greater than ten times the method detection limit constitute a failure.
- Duplicate samples with a difference greater than 10% are re-assayed unless the sample duplicate and original value are both less than 1 g/t.

Figure 12-1 shows the QA/QC validation flow charts for channel samples and drill core.

The treatment of drill-core CRM and channel sample CRM failures is equivalent. The failure is evaluated in the context of the surrounding samples and standards to determine if the failure is the result of a CRM or blank misidentification; if the failure is the result of a laboratory mix-up of samples or sample numbers; if the failure is the result of contamination; or if the failure is the result of other laboratory practices.

If it is possible to determine that the failure is the result of a CRM substitution error or a sample mix-up, either in submittal or at the lab, re-analyses of all or part of the batch may be requested.

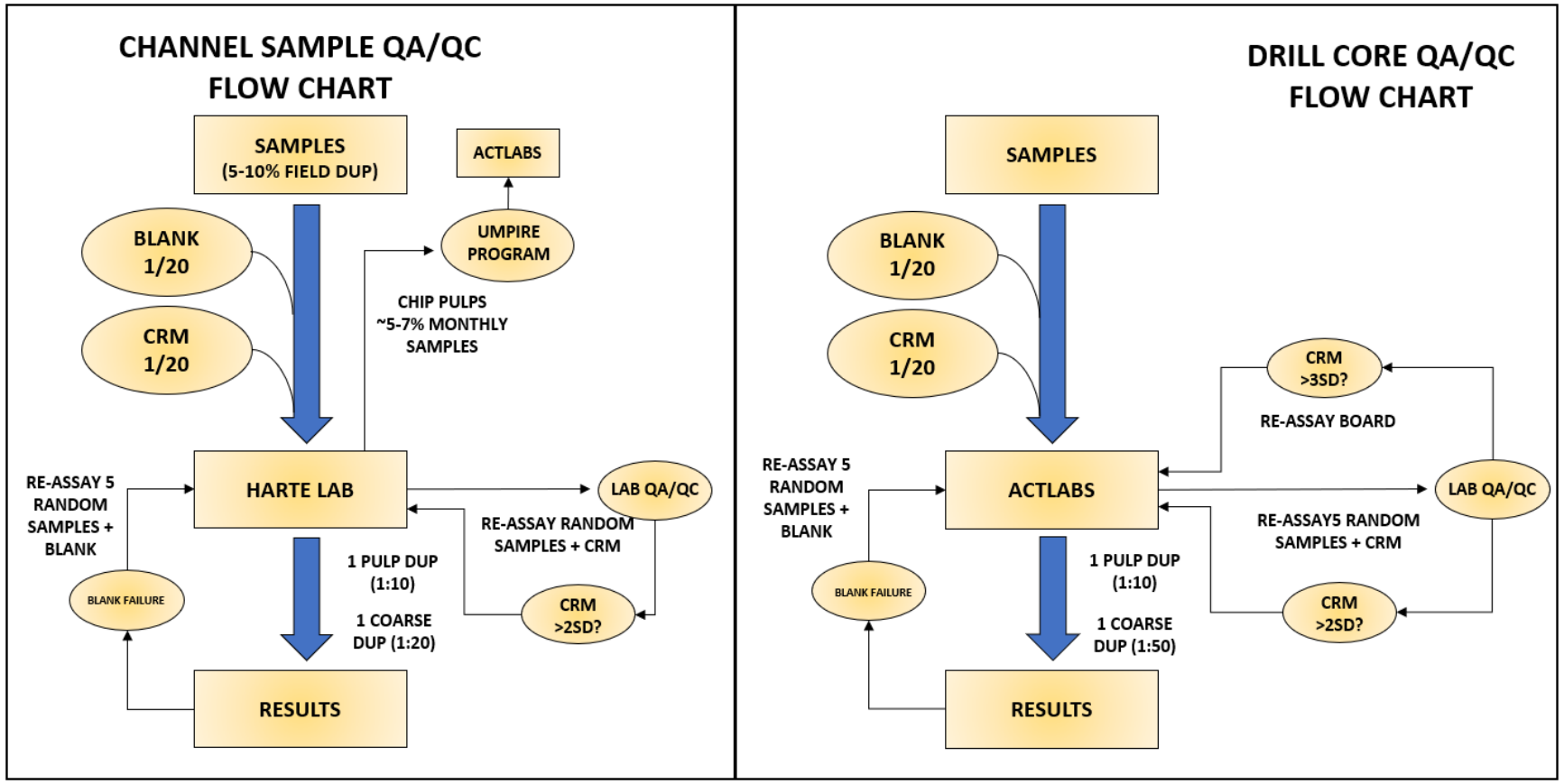


Figure 12-1: QA/QC Validation Flow Charts for Channel Samples and Drill Core

QA/QC data are reviewed on a continuous basis as assays are imported into their respective Excel spreadsheet. Harte Geology and Harte Assay Lab groups generate monthly QA/QC reports.

Monthly QA/QC reports include:

- A summary of samples shipped, and analyses received during the period
- Standard and blank performance summaries for the month and standard type
- Standard analysis bias summaries by standard type
- Individual CRM control plots for individual samples throughout the month
- Blank performance plots for samples throughout the month
- Duplicate scatterplots (umpire, coarse reject, and pulp reject)
- HARD plots for all duplicate types.

In all cases, Harte assay lab management provides insight into lab performance throughout the month.

12.2 Quality Assurance/Quality Control Analysis

The cut-off for the Mineral Resource estimate database was 31 March 2020. This was the same data set used in the QA/QC analysis described in the following sections.

12.2.1 Drill Core

12.2.1.1 Standards (Certified Reference Material)

In all, 1,389 CRM analyses have been conducted to date. This equates to roughly 5% of the total drill-core sample stream. Nine different CRMs were used during this period. The distribution and performance of each standard are presented by laboratory in Table 12-1 (Actlabs), Table 12-2 (AGAT), and Table 12-3 (Wesdome).

12.2.1.2 Actlabs (2010–2019)

Results from Actlabs (Table 12-1) show an overall slightly negative bias (−1.0%) by weight. The performance of the standards was reasonable, with only a handful of failures identified, some of which were the result of sample or CRM mix-ups (these were corrected whenever it was deemed reasonable). With that being said, the performance of STD 15PA was poor, with 60% of the assayed CRM failing. Use of this standard was discontinued at the end of 2010.

Table 12-1: Summary of Actlabs CRM Analyses from 2010–2019

Standard	Certified Mean	CRM Submitted	Mean Assay Grade	Laboratory Bias (%)
STD 10C	6.60	95	6.50	−2
STD 10PB	7.15	8	7.22	1
STD 15PA	1.02	10	1.09	7
STD 16A	1.81	88	1.82	1
STD 17C	3.04	44	3.05	0
STD 210	5.49	361	5.38	2
STD 215	3.54	319	3.52	1
STD 216	6.66	294	6.59	1
CDN-GS-3F	3.10	2	3.18	3

12.2.1.3 AGAT (2014)

Results from AGAT (Table 12-2) show an overall slightly positive bias (1.0%) by weight. The performance of the standards was reasonable, with only three failures occurring, one of which was sample mix-up with a blank.

Table 12-2: Summary of AGAT CRM Analyses from 2014

Standard	Certified Mean	CRM Submitted	Mean Assay Grade	Laboratory Bias (%)
STD 17C	3.04	123	3.06	1
STD 216	6.66	1	6.80	2

12.2.1.4 Wesdome (2015–2016)

Results from Wesdome (Table 12-3) show an overall slightly negative bias (−0.8%) by weight. The performance of these standards was reasonable. A handful of standards failed low, showing a clear preference for a negative bias.

Table 12-3: Summary of Wesdome CRM Analyses from 2015–2016

Standard	Certified Mean	CRM Submitted	Mean Assay Grade	Laboratory Bias (%)
STD 16A	1.81	6	1.75	−3
STD 17C	3.04	22	3.08	1
STD 216	6.66	16	6.48	−3

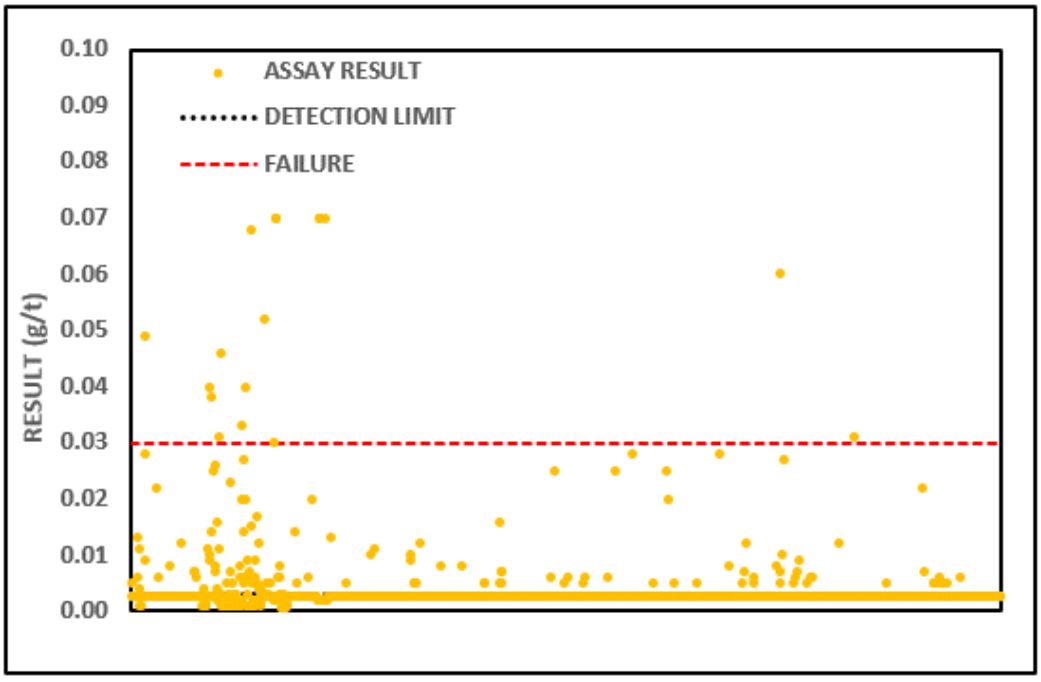
12.2.1.5 Blanks

A total of 1,660 blank analyses have been run since 2010. This represents approximately 5% of samples submitted in this period. Twenty-four failures (1%) were identified in the dataset, many of

which are less than 0.1 g/t. Three failures were identified (<1%) where the values returned were near 1 g/t or greater. Two of the failures returned values similar to the expected results of OREAS STD 215; it is possible that these were mixed up during shipment preparation. The third failure occurred just below 1 g/t. It is likely that this was either a sample mix-up (at the lab or in the field) or a contaminated sample. Table 12-4 presents a summary of blank analyses, and Figure 12-2 shows the control plot for this period.

Table 12-4: Summary of Blank Analyses from 2010–2019

Year	Count	Minimum (g/t)	Maximum (g/t)	Median (g/t)	Mean (g/t
2010	60	0.001	0.049	0.0025	0.005
2011	50	0.0025	0.216	0.0025	0.010
2012	17	0.025	0.007	0.0025	0.003
2013	32	0.001	0.104	0.0025	0.010
2014	90	0.001	0.238	0.0025	0.011
2015	49	0.0005	0.07	0.0025	0.007
2016	76	0.002	3.13	0.0025	0.087
2017	707	0.0025	0.992	0.0025	0.004
2018	495	0.0025	0.06	0.0025	0.003
2019	84	0.0025	0.006	0.0025	0.003



Note: Axis was constrained to 0.10 g/t for clarity

Figure 12-2: Blank Performance

12.2.1.6 Pulp Duplicates

Table 12-5 presents a summary of pulp duplicates from the 2018 and 2019 drill programs, which Actlabs performs at a frequency of 1 in 20 samples.

Table 12-5: Summary Statistics and HARD Results for 860 Paired Assay Results from the 2018 and 2019 Drill Programs

	Minimum Au (g/t)	Maximum Au (g/t)	Mean Au (g/t)	Variance	Standard Deviation	HARD			
						<10%	<20%	<30%	Correlation Coefficient
Original	0.00	53.10	0.24	4.2	2.06	711	811	865	1.00
Duplicate	0.00	53.10	0.23	4.0	2.01				

The HARD results (Figure 12-2) and scatter plot (Figure 12-3) for the paired duplicates are presented below. The HARD results show that approximately 711 pairs (83%) fall within 10% error. The correlation coefficient for these data is 1.0 and the R^2 value is 0.996. These results are adequate.

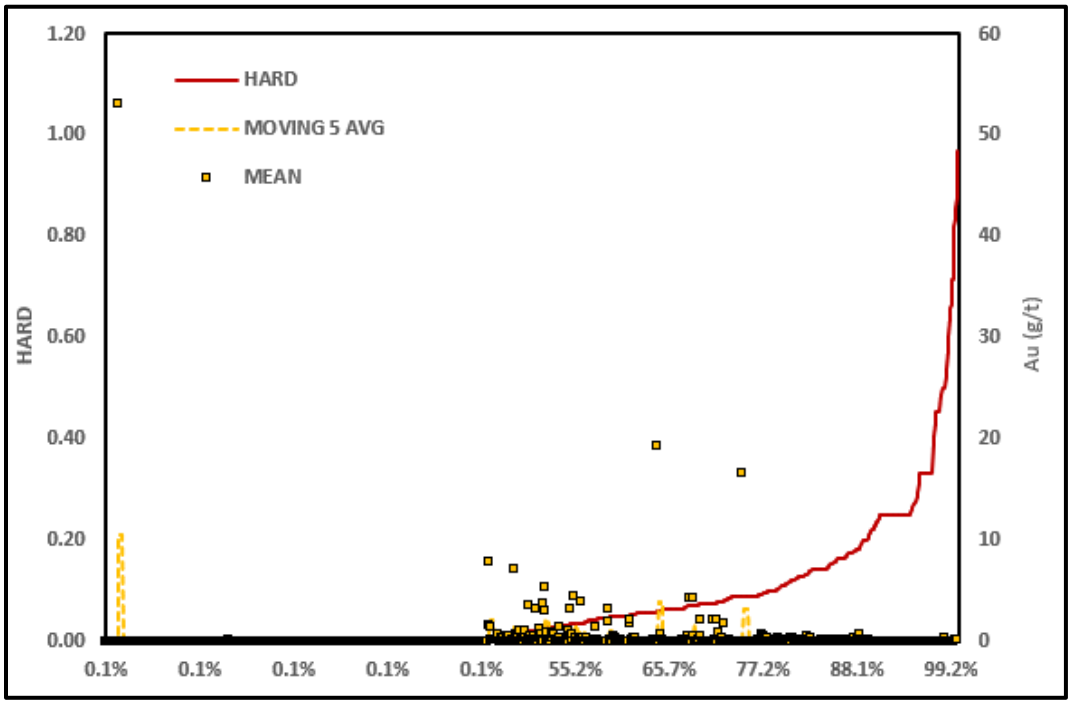


Figure 12-3: HARD Plot Showing HARD, Paired Mean Values, and 5 Points Moving Average

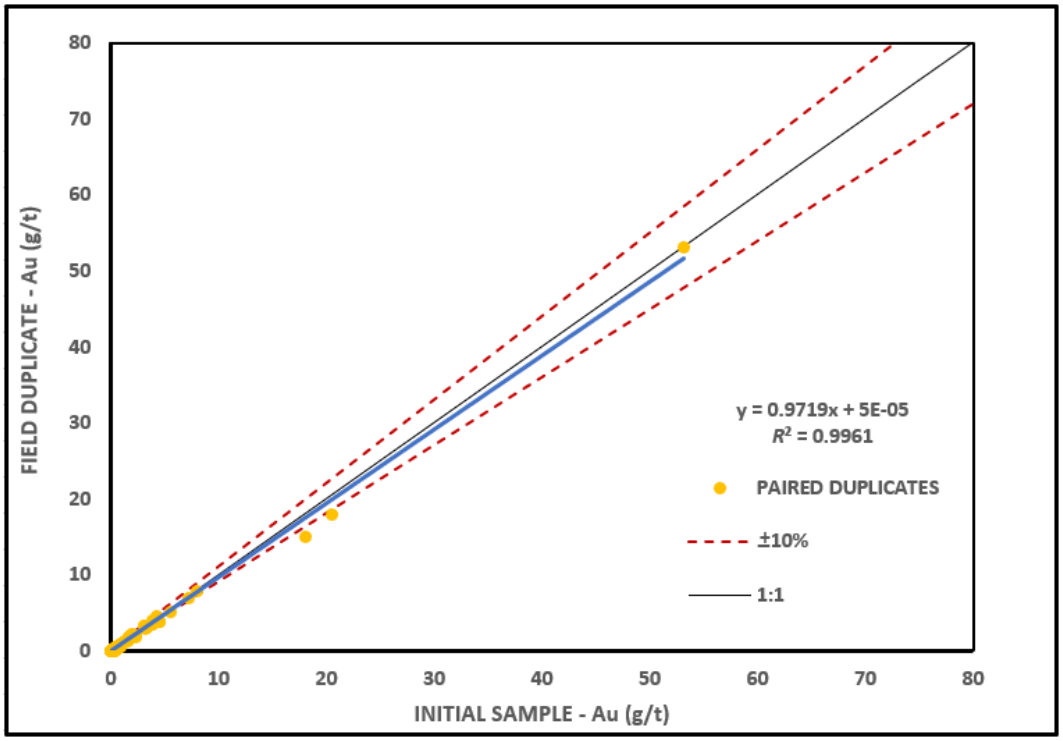


Figure 12-4: Paired Assay Results Showing 1:1 and ±10% Lines

12.2.1.7 Umpire Checks—P&E Due Diligence 2018–2019

P&E conducted due diligence checks on select core samples in January 2018, and again in January 2019. Each visit involved confirmation of surface drill-hole collar coordinates, a review of core logging procedures, and assessment of sampling protocols. A total of 28 field duplicates was collected over these periods. Sampling was conducted by quartering half-barrels of core. Samples from each visit were delivered to AGAT Laboratories for fire assay with atomic absorption or gravimetric finish, and specific gravity analysis.

P&E considered there was a good correlation between the original assay results and the field duplicate results. Grade variation is evident in some samples; however, this is within expected tolerances for a gold deposit. Figure 12-5 and Figure 12-6 show the results of the P&E field duplicates vs. the original assay results.

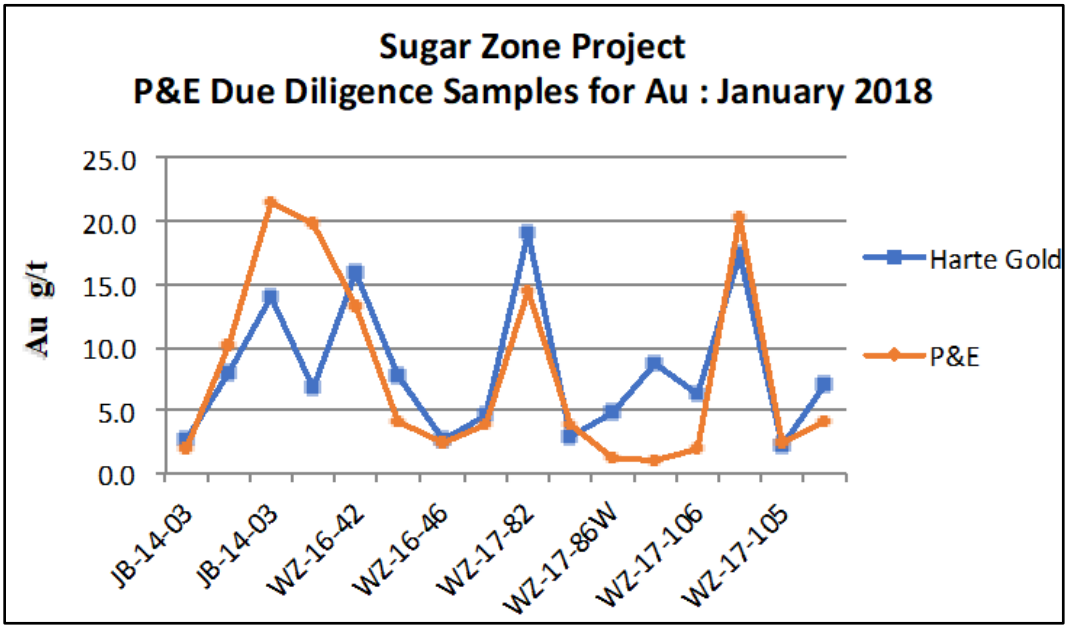


Figure 12-5: Results of Verification Sampling for Au by P&E—January 2018

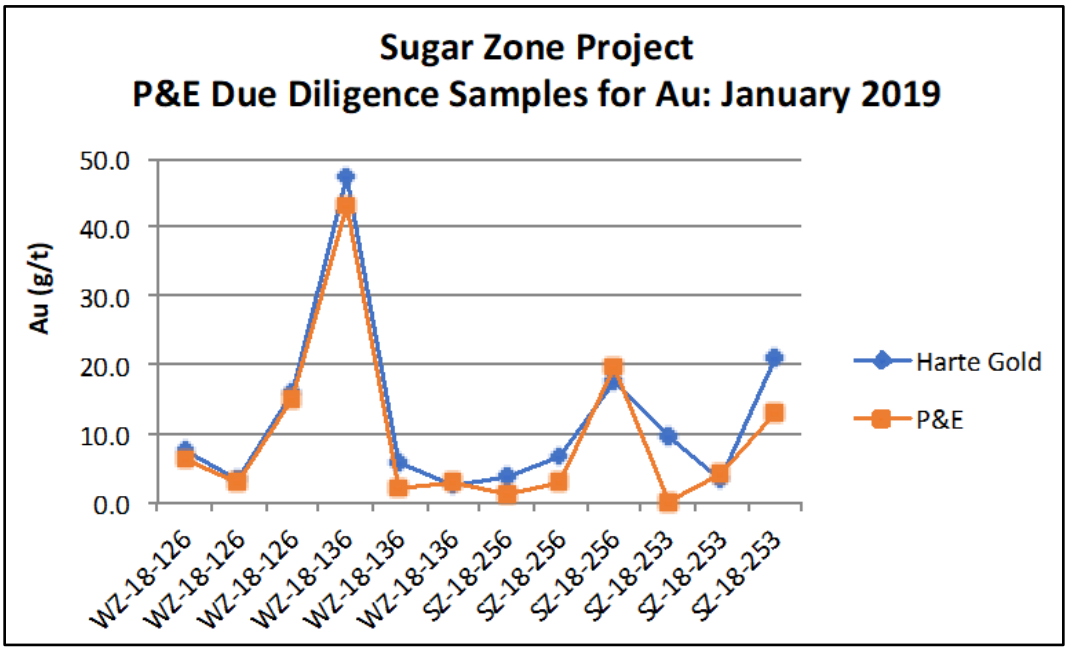


Figure 12-6: Results of Verification Sampling for Au by P&E—January 2019

As of 1 December 2020, Harte Gold has submitted 200 pulp rejects for umpire analysis to SGS Cochrane. Moving forward, Harte Gold will submit pulp rejects on a monthly basis, to represent approximately 5% of the total submitted samples.

12.2.2 Channel Samples

Data verification for the channel sample section is presented over two periods. Prior to 2019, grade control samples were delivered to the Wesdome Laboratory in Wawa, Ontario. Grade control analyses transitioned to the Harte site laboratory following commissioning in 2019.

12.2.2.1 Standards (Certified Reference Materials)

A total of 664 CRM analyses have been conducted to date. This equates to roughly 5% of the total channel sample stream. The distribution and performance of each standard are presented in Table 12-6. Five different OREAS CRMs were used during this period.

In general, standard performance has continued to improve since assaying moved to the Harte Laboratory.

12.2.2.2 Wesdome (2016–2018)

The Wesdome Laboratory (Table 12-6) tends to show a moderate positive bias versus the expected grade (0.60%–5.37%), with the exception of analyses of STD 10C, which shows a negative bias (−2.73%) versus the expected grade. This wide-ranging bias also explains the significant CRM failure rate (17% or 77 CRM) at this laboratory.

12.2.2.3 Harte Lab (2019–2020)

Standard performance improved once sample processing moved internally, with a failure rate of a more acceptable 4% (9 CRM). Results from the Harte lab (Table 12-6) show a negative bias (−1.95%) on the high-grade CRM while the other two CRMs perform (0.00% to 0.73%), as expected.

Table 12-6: Summary of Channel CRM by Laboratory from 2016–2020

Standard	Certified Mean (Au g/t)	Wesdome Laboratory (2016–2018)			Harte Laboratory (2019–2020)		
		CRM Submitted	Mean Assay Grade	Laboratory Bias (%)	CRM Submitted	Mean Assay Grade	Laboratory Bias (%)
STD 210	5.49	134	5.66	3.10	69	5.53	0.73
STD 215	3.54	85	3.73	5.37	67	3.54	0.00
STD 216	6.66	132	6.70	0.60	68	6.53	−1.95
STD 17C	3.04	94	3.17	4.28	0	-	-
STD 10C	6.60	23	6.42	−2.73	0	-	-

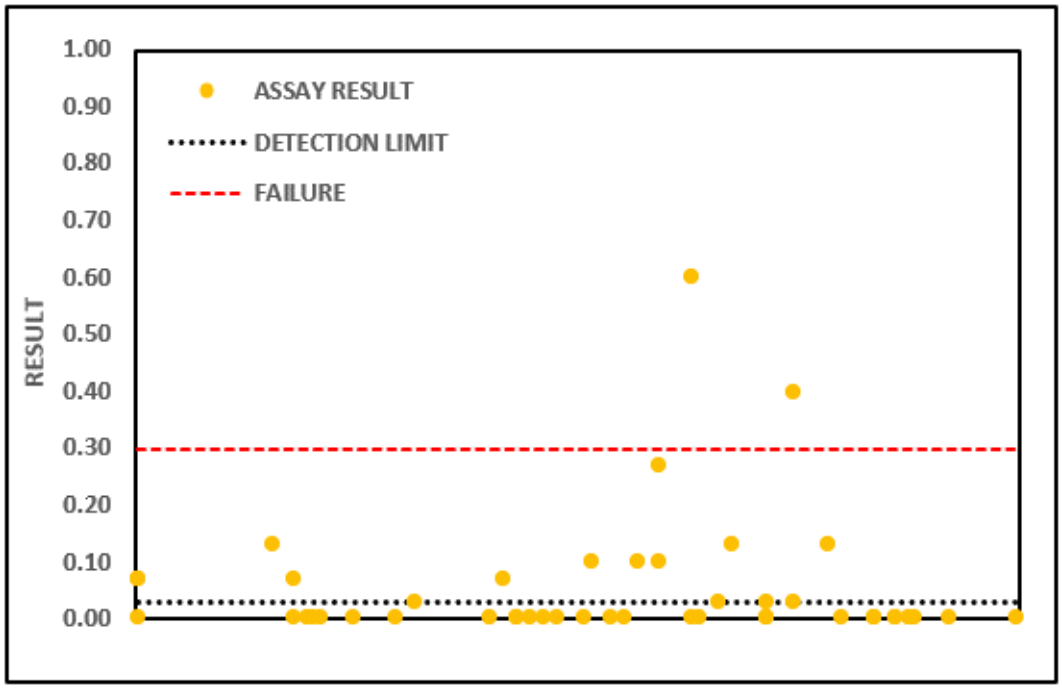
12.2.2.4 Blanks

As part of the QA/QC program put in place during the initial bulk sample between March and August 2016, 49 coarse blanks were submitted in 2016. This represents about 3% of submitted samples throughout this period. Four failures were identified from this period where the values returned were 10 times the detection limit (i.e., 0.3 g/t). Three of the four samples were re-assayed. Two of the re-

assays passed, with results less than the reported detection limit of the lab (0.03 g/t). There was a single significant failure of 6.0 g/t, which was re-assayed and returned a greater value of 9.27 g/t. It is likely that this was either a sample mix-up (at the lab or in the field) or a contaminated sample. Table 12-7 presents a summary of blank analyses, and Figure 12-7 shows the control plot for this period.

Table 12-7: Summary of Blank Analyses for 2016

Year	Count	Minimum (g/t)	Maximum (g/t)	Median (g/t)	Mean (g/t
2016	49	0.002	9.27	0.002	0.39



Note: Axis was constrained to 1.0 g/t for clarity.

Figure 12-7: Blank Performance

12.2.2.5 Pulp Duplicates

Wesdome (2016–2018)

Pulp duplicates (i.e., rejects) were analyzed at a frequency of 1 every 20 samples and when assay results were greater than 30 g/t.

The results are summarized in Table 12-8. Overall, there is a slight positive bias between the original and duplicate sample. HARD results (Figure 12-8) show that 1,063 duplicates of these samples (82%) fall within 10% precision, and 1,063 duplicates of the samples (98%) fall within 30% precision. The

scatter plot (Figure 12-9) shows an R^2 value of 0.9997, which is reasonable given the equipment used and the nature of mineralization.

Table 12-8: Summary Statistics and HARD Results for 1080 Paired Assay Results from the Wesdome Lab (2016–2018)

	Minimum Au (g/t)	Maximum Au (g/t)	Mean Au (g/t)	Variance	Standard Deviation	HARD			
						<10%	<20%	<30%	Correlation Coefficient
Original	0.00	456.37	38.94	3,683.7	60.69	890	1031	1063	0.98
Pulp Reject	0.00	480.33	39.15	3788.1	61.55				

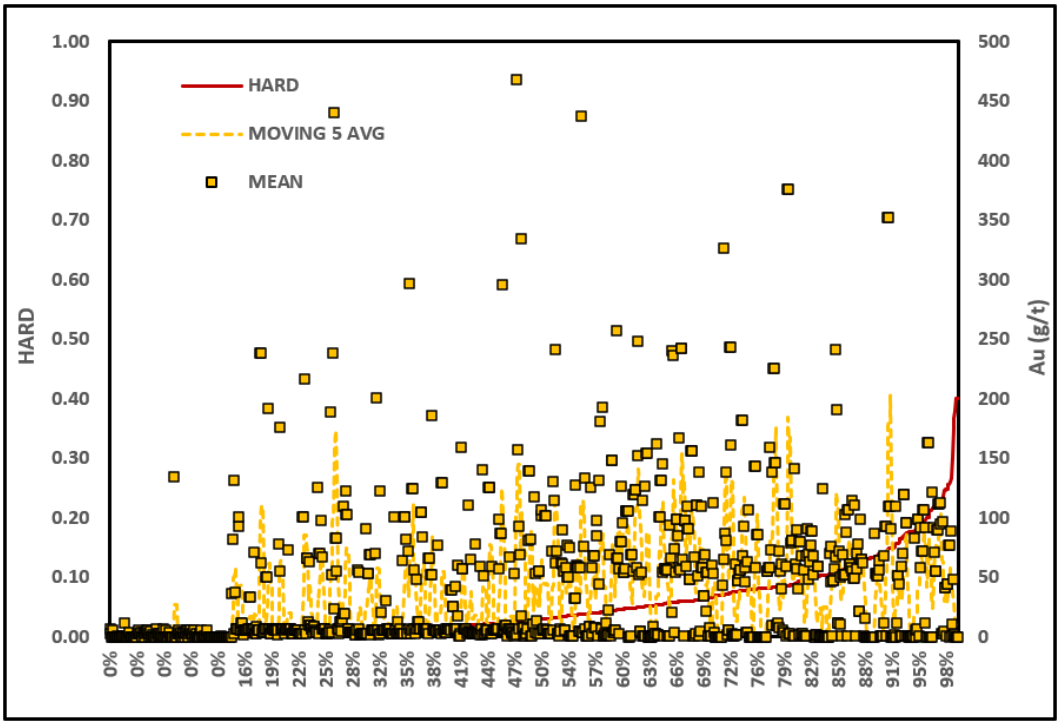


Figure 12-8: HARD Plot Showing HARD, Paired Mean Values, and 5 Points Moving Average

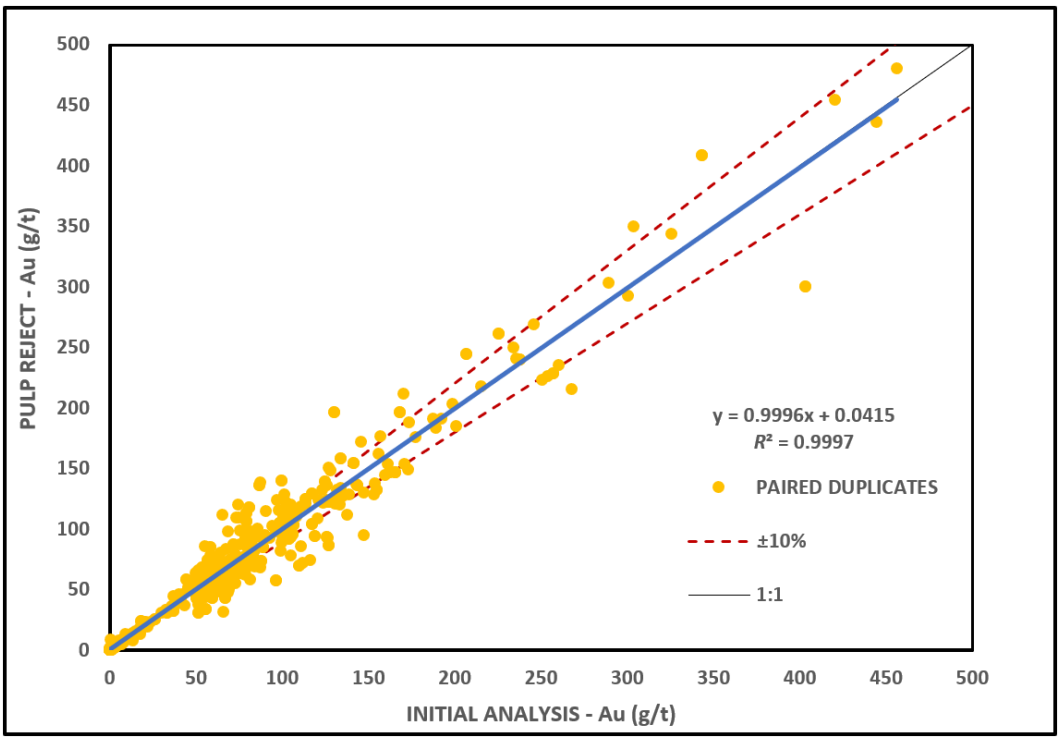


Figure 12-9: Paired Assay Results Showing 1:1 and ±10% Lines

Harte Lab (2019–2020)

Pulp duplicates are collected and processed at a frequency of 1 every 10 samples. Between May 2019 and March 2020, 293 paired channel assays were analyzed.

The results are summarized in Table 12-9. Overall, there is a slight negative bias between the original and duplicate sample. HARD results (Figure 12-10) show that 153 duplicates (52%) of these samples fall within 10% precision, and 243 duplicates (83%) of the samples fall within 30% precision. The scatter plot (Figure 12-11) shows an R^2 value of 0.972, which is reasonable given the equipment used and the nature of mineralization.

Table 12-9: Summary Statistics and HARD Results for 293 Paired Assay Results from the Harte Lab (2019–2020)

	Minimum Au (g/t)	Maximum Au (g/t)	Mean Au (g/t)	Variance	Standard Deviation	HARD			
						<10%	<20%	<30%	Correlation Coefficient
Original	0.00	166.38	5.53	330.3	18.17	153	217	243	0.99
Pulp Reject	0.00	162.79	5.46	314.4	17.73				

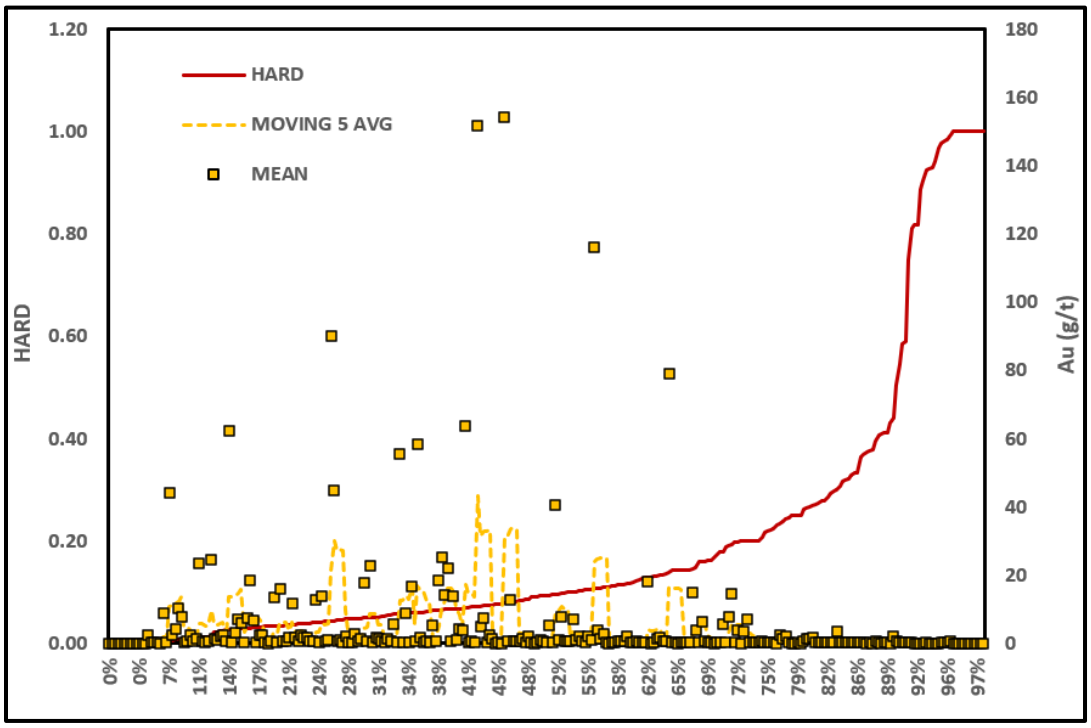


Figure 12-10: HARD Plot Showing HARD, Paired Mean Values, and 5 Points Moving Average

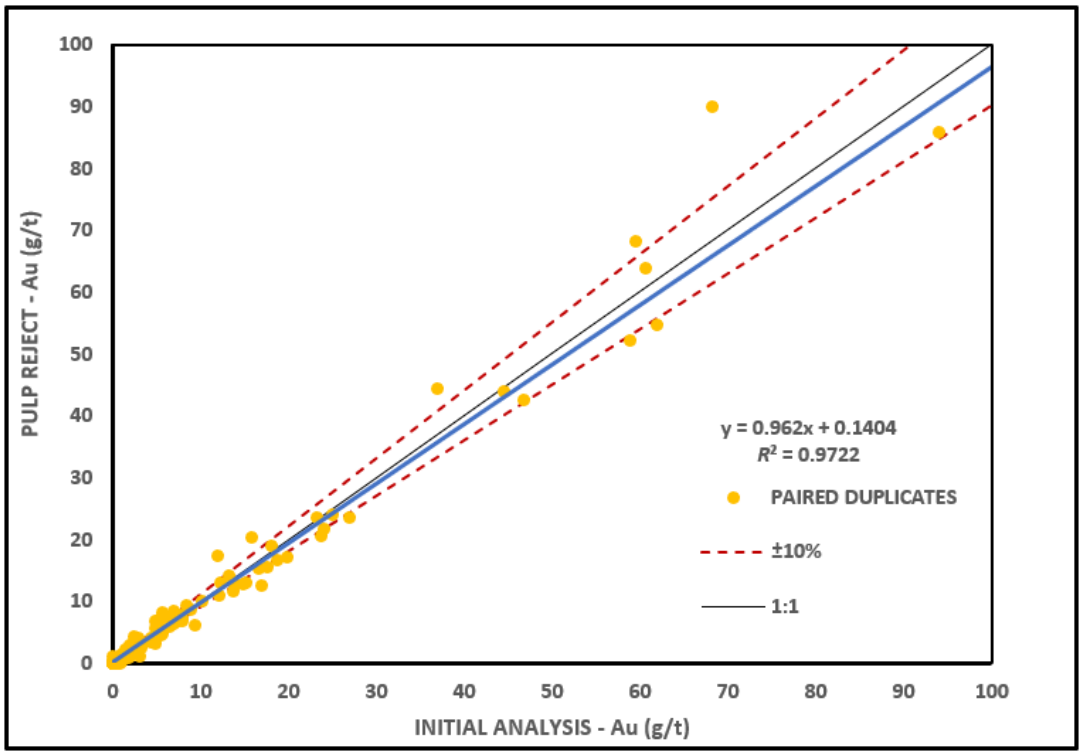


Figure 12-11: Paired Assay Results Showing 1:1 and ±10% Lines

Umpire Duplicates

Pulp rejects from the grade-control stream, muck, and channel samples have been submitted to Actlabs on a monthly basis since May 2019. From May 2019 to July 2020 a total of 5,001 channel samples were collected in the underground. The umpire samples presented below represent about 2% of the total samples collected over this period. To increase the umpire frequency, as of July 2020 only channel samples are being submitted for umpire checks, and the frequency has increased from 20 samples per month to 1 sample per day. This will represent roughly 4% to 5% of the channel sample stream.

Overall, the results suggest a lower-than-anticipated precision, at values <2 g/t, resulting in a negative bias versus the umpire analysis (Table 12-10). The cause of this is likely to the gravimetric method of analysis the Harte Laboratory employed. A summary of the paired samples is presented in Table 12-10. HARD results (Figure 12-12) show that 19 of the pairs (22%) fall within 10% precision and 50 pairs (58%) fall within 30% precision. The scatter plot (Figure 12-13) of paired assay values shows an R^2 value of 0.78, which is adequate.

Table 12-10: Summary Statistics and HARD Results for 86 Paired Assay Results

	Minimum Au (g/t)	Maximum Au (g/t)	Mean Au (g/t)	Variance	Standard Deviation	HARD			
						<10%	<20%	<30%	Correlation Coefficient
Original	0.00	61.45	6.65	180.5	13.44	19	35	50	0.89
Umpire	0.00	105.00	7.50	343.9	18.54				

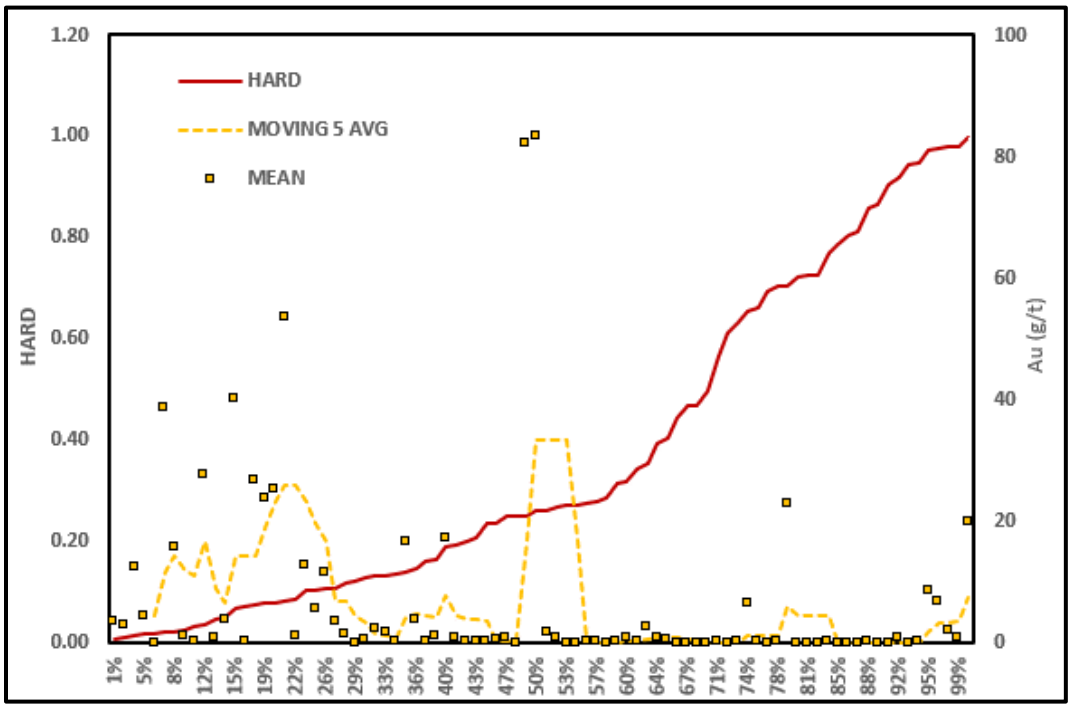


Figure 12-12: HARD Plot Showing HARD, Paired Mean Values, and 5 Mean Moving Average

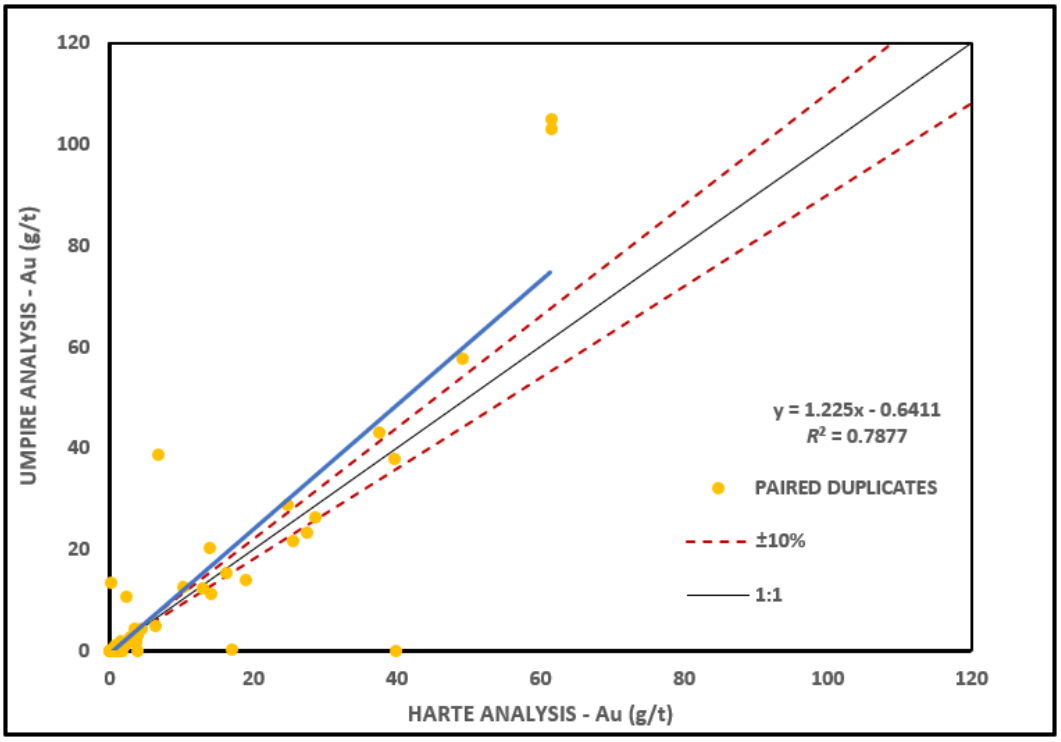


Figure 12-13: Paired Assay Results Showing 1:1 and ±10% Lines

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Historical Testwork Summary (2010–2017)

Up to the end of 2017, testwork was carried out on surface samples and drill core; it focused on determining metallurgical response and variability, by location and by gold and sulphide grades within the Sugar Zone and Middle Zone deposits.

The results from the four test programs carried out between 2010 and 2017 are summarized below, and in more detail in the following sections and the SGS reports. Taken together, the four programs metallurgically characterize the Sugar Zone deposit well.

A large number of samples from around the Sugar Zone deposit were tested for the four test programs, which included metallurgical, comminution, and environmental testing. Various composites were prepared from these samples, and variability testing was included in the final program. Gold head grades ranged from 1.6 g/t to 40 g/t Au. Silver head grades ranged from 0.8 g/t to 7.8 g/t Ag, and sulphur grades ranged from 0.3% to 0.9% S. Sulphur was present predominantly as the sulphide mineral, pyrrhotite.

The crusher work Index (CWi) fell in the moderately hard range of the SGS database. The Bond ball mill work Index (BWi) values were classified as medium hardness. The Bond abrasion index (Ai) values fell in the high end of the range of abrasiveness.

The gravity gold recovery was high for almost all samples, ranging from the mid-50s to over 90. These results indicated that a gravity circuit should be considered in the overall process flowsheet. High gold recoveries in the high 90% range were obtained from whole plant feed cyanidation at moderate grinds, and also by flotation of a rougher concentrate, at an approximate 4% to 5% mass pull. High gold recoveries were also obtained by cyanidation of the flotation concentrate.

Cleaner flotation test results indicated that flotation concentrate gold grades of greater than 100 g/t Au could be achieved at lower mass pulls of approximately 1% to 2%. Regrinding the rougher concentrate prior to cleaning also improved the concentrate gold grade and recovery.

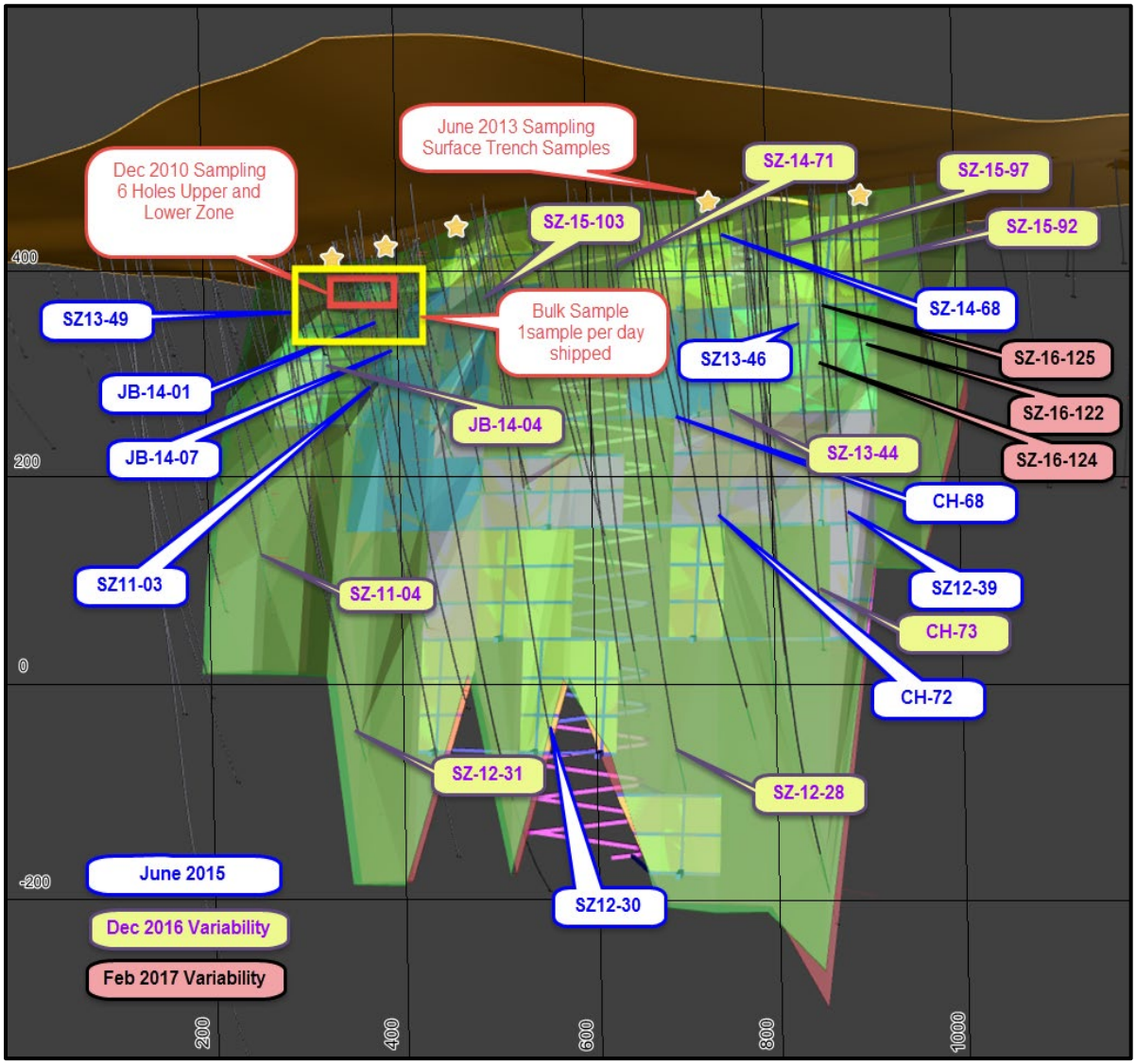
A flotation concentrate sample and a tailings sample were subjected to solid-liquid separation and filtration testwork. The samples responded well to BASF Magnafloc 10 flocculant, which is a very-high molecular weight, slightly anionic polyacrylamide flocculant. For both samples, pressure filtration resulted in lower residual cake moisture, while producing similar or higher solids throughput versus vacuum filtration.

Various flowsheets have been considered and tested during the four metallurgical test programs. The recoveries obtained by gravity and flotation only, compare well with whole plant feed cyanidation, with and without gravity.

A combined gravity-rougher flotation recovery of 96% is predicted. The likely gold split between the two steps is estimated to be 42% to doré and 54% to the flotation concentrate.

13.1.1 Sample Locations

The sample locations for all four test programs are shown in Figure 13-1. The Sugar Zone deposit outcrops close to surface at an elevation of 400 to approximately 430 masl. It is about 600 m deep and 800 m long.



Source: Harte Gold, 2018.

Note: Grid in metres.

Figure 13-1: Sample Locations

13.1.2 Testwork Up To 2016

There have been four sampling/metallurgical campaigns before 2016. The first three, 2010, 2013, and 2015, are briefly described below. Details are provided in SGS reports (Trang C.V., 2010, McCarley, 2013, Crary, 2016).

13.1.3 2010 Testwork Report Summary

The 2010 sampling was from a small area shown as the red box on Figure 13-1. A master composite was prepared by combining six pails of drill-core samples from the Upper and Lower Subzones, which assayed 14.7 g/t Au and 3.1 g/t Ag. A mineralogical study conducted on the composite sample indicated that it was composed mainly of quartz and other silicate minerals. Pyrrhotite was the most abundant sulphide mineral, while native gold and electrum were the most abundant gold minerals. Most of the gold grains (over 87%) were found to be liberated or exposed. The results suggested that the sample should respond well to gold recovery by gravity separation, flotation, and direct cyanidation leaching.

The sample BWi was 12.1 kWh/t.

SGS performed a standard gravity recoverable gold test on the sample. The gravity-recoverable gold was 89.1%. SGS conducted several gravity separation tests on the sample to generate products for downstream testing. Recovery was 77% to 90% for gold, and 48% to 59% for silver.

SGS completed a cyanidation test on the gravity tailing to evaluate gold recovery by cyanidation. Gold recovery was 85.6%, and silver recovery was 65.3%. The combined gravity plus leach recovery was 98.6% for gold and 85.8% for silver.

SGS conducted a further series of tests on the gravity tailing to investigate flotation gold recovery. Gold recovery to the rougher concentrate was 81% to 89% (combined gravity plus flotation gold recovery of 96% to 98%), at a mass pull of approximately 10%.

Regrinding the rougher concentrate to 80% passing (P_{80}) 30 µm, followed by cleaning, produced a cleaner concentrate assaying 70 g/t Au at a gold recovery of 79%. Overall gold recovery from flotation plus gravity was 96%.

SGS conducted a bulk flotation test on the gravity tailing to generate a rougher concentrate for gold leaching. It was possible to recover 98.6% of the gold and 83.8% of the silver by cyanidation of the reground rougher flotation concentrate at a P_{80} of 21 µm.

The testwork results suggested that producing a flotation concentrate required a flowsheet consisting of gravity separation followed by flotation of the gravity tailing. This option would allow sale of the flotation concentrate to a local process plant that can perform cyanidation in the treatment process. A second flowsheet option would be gravity separation followed by cyanidation of the gravity tailing.

13.1.4 2013 Testwork Report Summary

The 2013 sampling was taken from surface trenches; gold stars in Figure 13-1 indicate their approximate location. All sampling was from the Lower Zone in the Sugar Zone deposit.

Material was split out from each of seventeen bags and mixed to form the master comminution composite. Test results were obtained for the JK Tech semi-autogenous grinding (SAG) mill comminution (SMC), Ai, Bond rod mill index (RWi), and BWi tests. The SMC test composite was characterized as moderately soft with respect to resistance to impact breakage (A x b), and soft in terms of resistance to abrasion breakage (t_a). The composite was found to be highly abrasive, with an Ai of 0.616 g. The composite was categorized moderately hard, with an RWi of 11.5 kWh/t, and with a BWi of 16.1 kWh/t, also moderately hard.

Four composites were submitted for metallurgical testing of gravity and cyanidation flowsheets, without flotation. The main objective was to evaluate the effectiveness of a gravity separation circuit on gold recovery. The head assay results for the four metallurgical composites varied from 3.1 to 16.0 g/t Au. Gravity separation testing of the composites yielded excellent gold recovery results, between 67.8% and 83.8%.

The gravity tailing from each test was subjected to a 48-h standard cyanidation test. The combined process of gravity followed by carbon-in-leach (CIL), demonstrated good gold recovery results across the range of head grades and particle sizes tested, with gold recoveries ranging from 93.3% to 98.9%. As would be expected, it was demonstrated that the gold recovery increased with increasing gold head grade and finer grind sizes, to a P_{80} of 75 µm. Whole-plant feed cyanidation testing of the composites also yielded very good gold recovery results. However, a benefit was noticed when the gravity step was included, particularly for the lower-grade composite.

13.1.5 2015 Testwork Report Summary

The 2015 sampling was taken from the Lower Zone of the Sugar Zone deposit. A master composite was prepared from 13 holes distributed around the deposit. Hole locations are indicated in Figure 13-1. The objective of the program was to determine the response of a master composite to both a whole plant feed cyanidation and gravity/flotation/concentrate cyanidation flowsheet.

All but two of the samples were used for the master composite; one drill hole and the waste sample were not included. The individual drill-core gold head grades ranged from 1.59 g/t to 39.6 g/t Au, and the composite assayed 12.7 g/t Au. The individual samples and composite silver head grades ranged from 0.9 g/t to 8.2 g/t Au. The composite sulphur and sulphide sulphur head grades were 0.76% and 0.74%, respectively.

The whole-plant feed cyanidation test results indicated that high gold extractions, up to 98.8%, could be achieved at a finer feed size.

A large gravity separation test was completed, using the master composite. Gravity gold recovery was 66.4%. The gravity tailing was subjected to standard sulphide flotation test conditions and to provide samples for downstream leaching, mineralogy, and environmental testwork. The composite gold and sulphide recoveries were 89.1% and 84.4% at a mass pull of 3.3%. The combined gravity + flotation gold recovery was 96.4%.

Two flotation concentrate cyanidation tests were completed using the concentrate: the "as-is" sample and a reground sample. The concentrate produced from the master composite responded well to cyanidation, and after 48 h of leaching the gold leach extractions were between 97.4% and 99.1%. Regrinding the concentrate increased the overall gold extraction slightly, and slightly increased the cyanide and lime consumption.

The highest overall gold recovery was achieved using a whole-plant feed cyanidation flowsheet at a grind size of P_{80} of 74 µm. The gravity/flotation/cyanidation overall gold recoveries were slightly lower when compared to the whole-plant feed cyanidation results. However, it should be noted that the flotation tailing was not leached.

Bulk mineralogy testwork, which included automated rapid mineral scan (QEM-ARMS) and X-ray diffraction (XRD) analysis, was completed using the master composite flotation concentrate. The major minerals by mass percentage were pyrrhotite (~54%), plagioclase (~10%), amphibole/pyroxene (~10%), and quartz (~7%). The sample also contained about 2.5% pyrite.

13.1.6 Testwork Report Summary (2017)

The 2016/2017 testwork program is described below. Details are provided in SGS (Crary, 2017).

13.1.6.1 Sampling

A 70,000-tonne bulk sample was mined, sampled, and shipped to the nearby Hemlo Gold Mine for toll treatment using their cyanidation process. The location of this bulk sample is denoted by the yellow box in Figure 13-1 and includes the area sampled in 2010. Samples were taken daily, and were combined to represent a monthly period from May to December 2017. Exceptions are the May and June periods, which were combined. In this way, seven 50 kg samples were taken. Seven variability samples were taken from drill core as shown in Table 13-1.

Two master composite samples (monthly and variability) and a bulk monthly sample were created using the test charges for gravity separation, flotation, cyanidation, and solid-liquid separation testwork. The sample details and weights are outlined in Table 13-2.

The monthly composites and variability samples were submitted for the same head assays, which are presented in Table 13-3.

Table 13-1: Variability Sample Information

SGS Variability Sample ID	Hole-ID	Sample Type	From (m)	To (m)	Tag Number	Quadrant	Weight (kg)
Var 1	CH-73 SZ-12-28 SZ-13-44	Half core Half core Quarter core	347.2 506.0 184.2	348.6 507.8 186.0	423902 423903 423904	Lower South Lower South Lower South	8.8
Var 2	SZ-14-71 SZ-15-92 SZ-15-97	Quarter core Quarter core Half core	25.8 79.1 25.5	28.6 81.5 27.7	423905 423906 423911	Upper South Upper South Upper South	8.9
Var 3	SZ-11-04 SZ-12-31	Half core Quarter core	242.0 468.0	243.8 470.2	423907 423909	Lower North Lower North	6.5
Var 4	SZ-15-103 JB-14-04	Half core Quarter core	94.0 146.6	96.0 150.1	423908 423910	Upper North Upper North	7.7
Var 5	SZ-16-122	Quarter core	186.1	188.1	1370345	Upper South	1.7
Var 6	SZ-16-124	Quarter core	149.5	151.4	1370346	Upper South	1.6
Var 7	SZ-16-125	Quarter core	137.9	138.8	1370347	Upper South	1.4

Table 13-2: Master Composites and Bulk Monthly Sample and Test Details

Sample Name	Samples	Weight (%)	Total Weight (kg)	Used For
Monthly Master Composite	May/Jun	20	20	Two 10 kg tests Gravity + Flotation + Cyanide Flotation (No Gravity)
	Jul	20		
	Aug	20		
	Nov	20		
	Dec	20		
Variability Master Composite	Var 1	25	10	One 10 kg test Gravity + Flotation + Cyanide
	Var 2	25		
	Var 3	25		
	Var 4	25		
Monthly Bulk Sample	May/Jun	60	160	Bulk (150 kg) Gravity + Flotation test to create flotation concentrate sample for solid/liquid separation testwork. One 10 kg Gravity + Flotation test to create flotation concentrate for analysis at a refinery
	Aug	13.3		
	Sep	13.3		
	Oct	13.3		

Table 13-3: Head Analysis Summary

Element	Unit	Monthly Composites							Variability Samples						
		May/Jun	Jul	Aug	Sep	Oct	Nov	Dec	Var 1 (LS)	Var 2 (US)	Var 3 (LN)	Var 4 (UN)	Var 5 (US)	Var 6 (US)	Var 7 (US)
Au Cut A	g/t	6.64	6.34	7.01	6.26	8.45	6.89	4.15	3.72	5.37	3.81	3.79	7.22	9.92	5.60
Au Cut B	g/t	7.92	6.00	5.81	4.88	31.4	6.34	3.08	2.40	9.27	2.99	4.63	6.21	11.3	4.86
Au Avg.	g/t	7.28	6.17	6.41	5.57	19.9	6.62	3.62	3.06	7.32	3.40	4.21	6.72	10.6	5.23
Au Calc.	g/t	7.28	5.87	7.45	6.99	7.98	6.84	5.48	3.50	5.40	6.11	3.42	4.36	18.2	6.60
Ag	g/t	1.9	2.0	1.8	1.9	1.8	3.0	1.2	1.3	1.3	1.1	0.8	1.6	7.8	2.7
Cu	%	0.014	<0.01	0.010	0.013	<0.01	<0.01	0.010	0.030	0.022	<0.01	0.014	0.020	0.016	0.018
Fe	%	4.40	3.56	4.77	4.74	3.91	3.95	4.24	5.83	3.76	5.22	3.21	7.38	6.12	5.77
S	%	0.75	0.76	0.90	0.94	0.75	0.82	0.76	0.84	0.70	0.35	0.70	0.48	0.46	0.57
S=	%	0.63	0.61	0.75	0.79	0.65	0.67	0.60	0.74	0.62	0.32	0.60	0.35	0.42	0.53

Notes: LS = Lower South; US = Upper South; LN = Lower North; UN = Upper North

13.1.6.2 Comminution

The existing comminution circuit at Harte Gold consists of primary and secondary crushing, followed by a ball mill. Therefore, comminution testing was limited to the CWi, BWi, and Ai.

Table 13-4 summarizes the Bond low-energy impact test results. The CWi fell in the moderately hard range of the SGS database, with a value of 12.7 kWh/t.

Table 13-4: Summary of Bond Low-Energy Impact Test Results

Sample Name	Number of Specimens	Work Index (kWh/t)	Min. (kWh/t)	Max. (kWh/t)	SD (kWh/t)	Relative Density
Harte Gold Sample	6	12.7	4.7	19.2	5.2	2.78

Note: SD = Standard Deviation.

The Bond ball mill grindability test was performed at a closing screen size of 75 µm on the seven monthly composites. The test results are summarized in Table 13-5. BWi values were similar for the samples and ranged from 14.2 to 15.0 kWh/t, averaging 14.7 kWh/t. All the samples were classified as medium hardness compared to the SGS database.

Table 13-5: Bond Ball Mill Grindability Test Results

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)
May/Jun Composite	200	2,430	63	1.22	14.6
Jul Composite	200	2,360	62	1.18	14.9
Aug Composite	200	2,346	61	1.20	14.6
Sep Composite	200	2,284	62	1.26	14.2
Oct Composite	200	2,264	60	1.15	15.0
Nov Composite	200	2,238	62	1.19	14.9
Dec Composite	200	2,277	60	1.15	15.0

Notes: F_{80} = Feed Size; P_{80} = Grind Size.

The seven monthly composites were submitted for the Ai test; results are summarized in Table 13-6. The Ai values ranged from 0.563 to 0.677 g, and all fell in the abrasive range of the SGS database.

Table 13-6: Bond Abrasion Test Results

Sample Name	Ai (g)
May/Jun Composite	0.563
Jul Composite	0.587
Aug Composite	0.601
Sep Composite	0.665
Oct Composite	0.677
Nov Composite	0.652
Dec Composite	0.631

13.1.6.3 Gravity and Rougher Flotation

A summary of the results from the gravity and flotation tests is given in Table 13-7.

Table 13-7: Gravity Concentration and Flotation Test Results Summary

Sample	Product	Weight %	Assays		Distribution %			
					Au			
			Au g/t	S %	Flot (Unit)	Grav	Grav+Flot	S
May/June	Rougher Conc 1-5	3	31.8	21.1	83.4	84	97.3	89.3
July	Rougher Conc 1-5	3.8	65.5	17.9	93.5	54.6	97.1	94.6
Aug	Rougher Conc 1-5	3.3	51.1	22.6	82.7	72.3	95.2	90.7
Sept	Rougher Conc 1-5	3.5	36.7	23.5	86.3	78.9	97.1	91.3
Oct	Rougher Conc 1-5	2.8	51.4	22.8	86.8	78.9	97.2	90.5
Nov	Rougher Conc 1-5	3.3	59.7	21.3	91.1	68.4	97.2	93.6
Dec	Rougher Conc 1-5	3.1	56.1	20.7	90.5	64.7	96.6	91.8
Monthly Master	Rougher Conc 1-5	3.4	55.2	19.9	92	68.7	97.5	90.9
Bulk Monthly	Rougher Conc 1-5	2.7	36.1	24.3	80.8	82.3	96.6	87.2
Var 1	Rougher Conc 1-5	4.1	20.1	19.4	69.4	66	89.6	85.6
Var 2	Rougher Conc 1-5	3.3	30.3	20.9	57.1	67.4	86	90
Var 3	Rougher Conc 1-5	2.8	77.6	12.3	96.6	63.1	98.7	92.2
Var 4	Rougher Conc 1-5	3.8	33.1	18.2	85.6	57.1	93.8	85.6
Average		**3.3**	**46.5**	**20.4**	**84.3**	**69.7**	**95.4**	**90.3**

For the monthly composites, the overall gravity + flotation gold recoveries ranged from 95.2% to 97.5%, and averaged 96.9%. The gravity gold recovery values for the tests completed using the monthly composites ranged from 54.6% to 84.0%, and averaged 73.3%. The results illustrated that the overall gold recoveries where essentially unaffected by changes in head grade.

Regarding the variability samples, the F-1 (Var 1) and F-2 (Var 2) flotation gold recovery values were 69.4% and 57.1%, well below the average of the other four variability tests. To resolve this, two additional variability samples from the Sugar Zone's Upper South Subzone were tested (F-16 to F-18) to confirm whether the poor flotation responses witnessed in tests F-1 and F-2, were related to that area of the deposit. Ignoring the poor results from F-1 and F-2, the results from the other six tests showed that, for the variability samples, the overall combined gravity and flotation gold recoveries ranged from 93.8% to 98.7% and averaged 97.1%. It was therefore assumed that variability Samples 1 and 2 were partially oxidized, due to their long period in storage.

Similar to the monthly composite results, the variability samples had high gravity gold recoveries that ranged from 57.1% to 90.1%, and averaged 74.8%.

Although only a single test was carried out—recovery from flotation only—the result was similar to recovery from combined gravity and flotation, suggesting that the combined gravity/flotation flowsheet is the correct type to stay with.

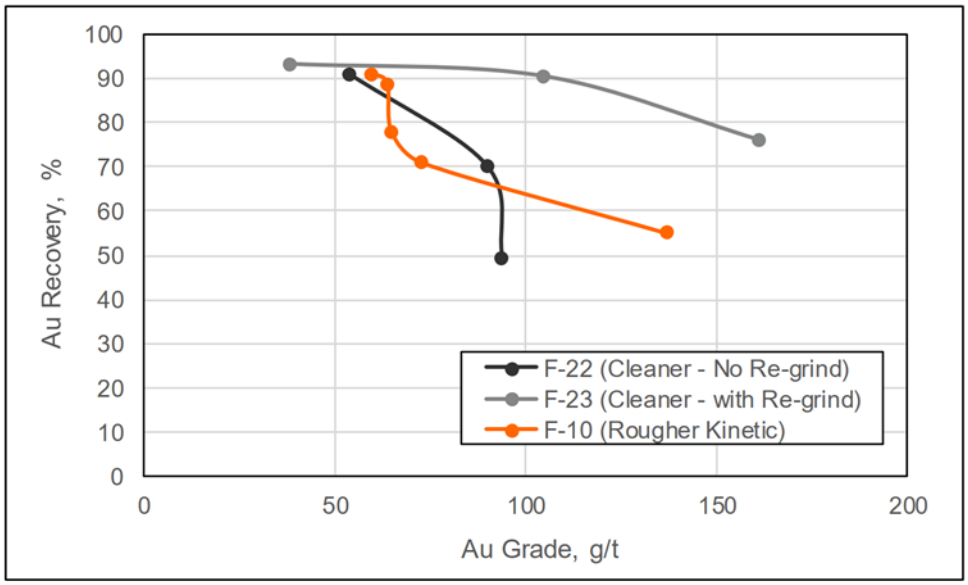
13.1.6.4 Regrind and Cleaner Flotation

The SGS test results are summarized in Table 13-8. The gold and sulphide grade/recovery curves are shown in SGS (Crary, 2017). Only two batch rougher and cleaner flotation tests were carried out, F-22 and F-23. F-22 was without regrind, and F-23 was with regrind. It can be seen that the test with regrind produced a higher-grade cleaner concentrate at a lower mass pull, with higher gold and lower sulphide recovery.

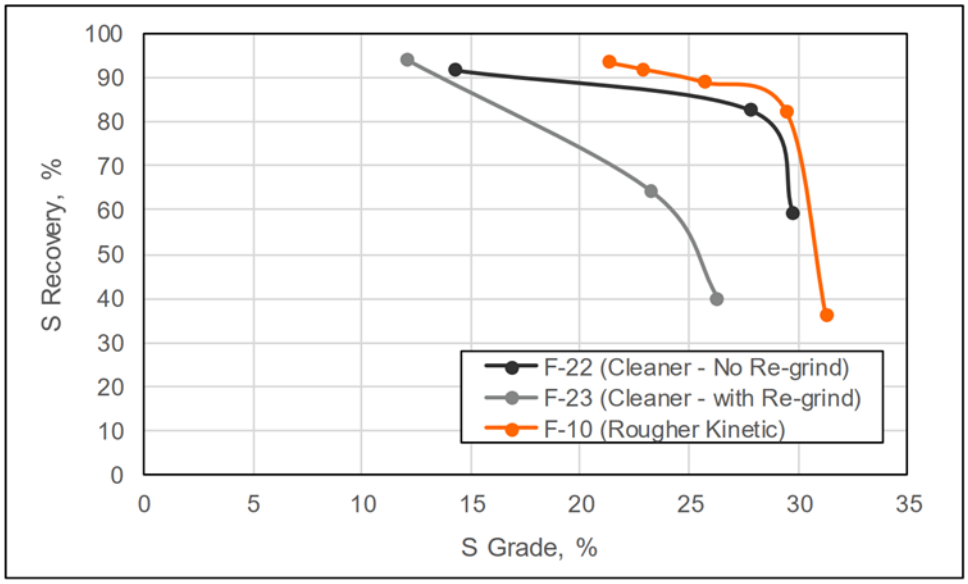
Table 13-8: Cleaner Flotation Results

Sample	Text No.	Test Size (kg)	P_{80}, µm				Reagents	Product	Weight (%)	Assay		% Distribution		
			Rougher Tailings	1st Cl. Conc. 1	1st Cl. Conc. 2	Cl. Conc.[1]				Au (g/t)	S (%)	Au		S
												Flot	Grav+Flot	
Nov	F-22 (G-20)	2	81	89	80	86	65 g/t PAX	**Mozley Conc.**					**39.8**	
							20 g/t 407	1st Cleaner Conc. 1	1.6	93.8	29.8	49.4	69.6	59.6
							30 g/t MIBC	1st Cleaner Con. 1-2	2.4	89.7	27.9	70.3	82.1	82.7
								Rougher Conc 1-5	5.2	53.7	14.3	91.1	94.6	91.9
							30 g/t PAX	Rougher Tail	94.8	0.29	0.07	8.9	5.4	8.1
							(cleaner)	Head (calculated)	100.0	3.09	0.81	100.0	100.0	100.0
Nov	F.23 (G-21)	2	81	28	20	24	65 g/t PAX	**Mozley Conc.**					**52.5**	
							20 g/t 407	1st Cleaner Conc.1	1.2	161	26.3	76.3	88.7	39.8
							25 g/t MIBC	1st Cleaner Conc. 1-2	2.2	105	23.2	990.5	95.5	64.2
								Rougher Conc. 1-5	6.1	38.3	12.1	93.2	96.8	94.0
							30 g/t PAX	Rougher Tail	93.9	0.18	0.05	6.8	3.2	6.0
							(cleaner)	Head (calculated)	100.0	2.49	0.78	100.0	100.0	100.0

Note: [1] Calculated (weighted average of Cleaner Concentrate 1 and 2).
Cl. Conc. = cleaner concentrate.



Gold Recovery vs. Gold Grade (Cleaner Tests)



Sulphur Recovery vs. Sulphur Grade (Cleaner Tests)

Source: Trang C.V.

Figure 13-2: Gold and Sulphur Recoveries

It was also demonstrated that it was possible to upgrade the rougher concentrate, after regrinding to 30 µm, in a single cleaning stage (Trang C.V., 2010).

The gold recovery from the rougher concentrate to 1st Cleaner Concentrate 1 only, is low, whereas the gold recovery to 1st Cleaner Concentrate 1 and 2 combined is much higher. That is, the overall gold losses to cleaner tails are 1.3% lower after removal of 1st Cleaner Concentrate 1 and 2. In the

actual plant, the cleaner tails are recirculated to the head of the rougher, where typically at least 50% is estimated to be recovered. Therefore, an overall gold loss on upgrading by cleaner flotation of approximately 0.7% is predicted. However, only a locked cycle test, or actual plant operation could accurately quantify this.

13.1.6.5 Concentrate Testing

As the flotation concentrate would typically be sold to a smelter, detailed minor element analyses were carried out. The results are presented in Table 13-9. The concentration of deleterious elements such as arsenic, antimony, and bismuth is generally low.

Table 13-9: Flotation Concentrate Analysis

Element	Rougher Concentrate		Element	Rougher Concentrate	
	F-12 (Monthly)	F-13 (Var)		F-12 (Monthly)	F-13 (Var)
Au g/t	55.2	26.6	Mg g/t	8,740	7,800
S %	19.9	21.1	Mn g/t	578	596
Si %	12.3	11.5	Mo g/t	<50	<50
TOC %	0.40	0.25	Na g/t	7,210	5,790
Ag g/t	30.6	17.6	Ni g/t	793	1,074
Al g/t	33,300	29,500	P g/t	<300	<300
As g/t	52	<30	Pb g/t	4,040	4,730
Ba g/t	136	150	Sb g/t	<10	<10
Be g/t	<1	<1	Se g/t	<30	<30
Bi g/t	<20	<20	Sn g/t	<30	<30
Ca g/t	31,200	33,000	Sr g/t	90.3	88.9
Cd g/t	63	102	Ti g/t	1,980	1,760
Co g/t	308	351	Tl g/t	<30	<30
Cr g/t	126	279	U g/t	<50	<50
Cu g/t	2,150	3,360	V g/t	78	81
Fe g/t	329,000	340,000	Y g/t	6.3	6.1
K g/t	8,910	7,220	Zn g/t	5,920	9,440
Li g/t	<5	<5			

Note: F-12 and F-13 are concentrates produced from the monthly and variability master composite gravity/flotation test. Monthly = monthly test; Var = variability test.

13.1.6.6 Whole-Plant Feed Cyanidation

The seven monthly composite samples were submitted for standard whole-plant feed cyanidation tests of mineralization. Gold extractions averaged 97.9% after 48 h of leaching. The cyanide and lime consumptions averaged 0.94 kg/t NaCN and 0.67 kg/t CaO (kg/t of CN feed). The results are summarized in Table 13-10.

Table 13-10: Whole Plant Feed Cyanidation Summary

Monthly Comp	CN Test No.	Feed Size (P_{80}, µm)	Reagent Addition (kg/t of CN Feed)		Reagent Cons. (kg/t of CN Feed)		Au Extraction (%)			Au Residue (g/t)			Au Head (g/t)	
			NaCN	CaO	NaCN	CaO	8 h	24 h	48 h	A	B	Avg.	Calc.	Direct
May/Jun	1	70	1.69	0.79	1.16	0.75	21	64	98.2	0.18	0.09	0.14	7.51	7.28
Jul	2	71	1.58	0.69	1.03	0.65	22	69	98.5	0.08	0.09	0.09	5.84	6.17
Aug	3	71	1.58	0.71	1.01	0.68	25	69	98.1	0.11	0.13	0.12	6.19	6.41
Sep	4	70	1.58	0.72	0.99	0.69	22	62	95.6	0.43	0.21	0.32	7.29	5.57
Oct	5	71	1.58	0.73	1.02	0.69	26	77	98.5	0.09	0.09	0.09	6.21	19.9
Nov	6	70	0.94	0.61	0.35	0.59	62	97	98.6	0.08	0.08	0.08	5.91	6.62
Dec	7	71	1.60	0.70	1.02	0.67	28	74	98.1	0.09	0.09	0.09	4.86	3.62
Minimum		**70**	**0.94**	**0.61**	**0.35**	**0.59**	**21**	**62**	**95.6**	**-**	**-**	**0.08**	**4.86**	**3.62**
Maximum		**71**	**1.69**	**0.79**	**1.16**	**0.75**	**62**	**97**	**98.6**	**-**	**-**	**0.32**	**7.51**	**19.9**
Average		**71**	**1.51**	**0.71**	**0.94**	**0.67**	**29**	**73**	**97.9**	**-**	**-**	**0.13**	**6.26**	**7.94**

13.1.6.7 Flotation Concentrate Cyanidation Testwork

Three flotation concentrate cyanidation tests were completed using the concentrate produced from the monthly and variability master composite gravity/flotation tests (F-12 and F-13). The F-12 concentrate was subjected to two tests (with and without regrind). The F-13 concentrate was not reground prior to leaching. A laboratory pebble mill was used to regrind the flotation concentrate prior to leaching. The cyanidation test conditions are shown in Table 13-11.

Table 13-11: Cyanidation Test Conditions

Description	Parameters
Charge Size	~120–150 g
Pulp Density	20% solids (w/w)
Pulp pH	10.5–11 maintained with lime
Cyanidation Concentration	1 g/L NaCN (maintained)
Retention Time	48 h (subsamples at 8 and 24 h)
Dissolved Oxygen Concentration	~20–25 mg/L target (oxygen added @ 0.02 L/min)

The tests were conducted in standard bottle rolls, and subsamples of the leach solution were taken at specific time intervals and assayed for gold to monitor the rate of extraction. The flotation concentrates responded well to cyanidation, and after 48 h of leaching the gold extractions were about 96% to 99%. Regrinding the concentrate produced from the monthly master composite increased the overall gold extraction slightly, and increased cyanide and lime consumption.

13.1.6.8 Other Testwork

Flotation concentrate and tailing samples were subjected to solid–liquid separation testwork. The samples responded well to BASF Magnafloc 10 flocculant, which is a very-high molecular weight, slightly anionic polyacrylamide flocculant. Optimized dynamic thickening test conditions of the monthly flotation tailing composite sample indicated that it settled well at 15% w/w solids feedwell

density and at a dosage of approximately 20 g/t flocculant. Optimized two-stage static settling test results indicated that the flotation concentrate sample settled well at 10% w/w solids and at a dosage of 25 g/t flocculant. For both samples, pressure filtration resulted in lower residual cake moisture while producing similar or higher solids throughput versus vacuum filtration.

13.1.6.9 Environmental Testwork

A bulk gravity/rougher flotation test (37 kg) was completed using several monthly composite samples to produce representative flotation tailing sample F-24, for environmental analysis, see Table 13-12.

Table 13-12: F-24 Environmental Tailing Solution Analysis

Analysis	Unit	F-24	Analysis	Unit	F-24
Temperature Upon Receipt	°C	20.0	Iron (total)	mg/L	0.369
pH	no unit	7.93	Iron (dissolved)	mg/L	0.106
Alkalinity	mg/L as $CaCO_3$	56	Potassium (total)	mg/L	13.4
Conductivity	µS/cm	291	Potassium (dissolved)	mg/L	12.8
Total Suspended Solids	mg/L	2	Magnesium (total)	mg/L	2.42
Ammonia+Ammonium (N)	as N mg/L	1.1	Magnesium (dissolved)	mg/L	2.26
Sulphate	mg/L	44	Manganese (total)	mg/L	0.0225
Chloride	mg/L	27	Manganese (dissolved)	mg/L	0.0192
Nitrite (as N)	as N mg/L	0.09	Molybdenum (total)	mg/L	0.0114
Nitrate (as N)	as N mg/L	1.59	Molybdenum (dissolved)	mg/L	0.00400
Hardness	mg/L as $CaCO_3$	63.8	Sodium (total)	mg/L	23.6
Hardness (dissolved)	mg/L as $CaCO_3$	60.6	Sodium (dissolved)	mg/L	22.7
Silver (total)	mg/L	0.00006	Nickel (total)	mg/L	<0.0001
Silver (dissolved)	mg/L	<0.00005	Nickel (dissolved)	mg/L	<0.0001
Aluminum (total)	mg/L	0.210	Phosphorus (total)	mg/L	<0.003
Aluminum (dissolved)	mg/L	0.052	Phosphorus (dissolved)	mg/L	0.063
Arsenic (total)	mg/L	0.0043	Lead (total)	mg/L	0.00385
Arsenic (dissolved)	mg/L	0.0032	Lead (dissolved)	mg/L	0.00006
Barium (total)	mg/L	0.00433	Antimony (total)	mg/L	0.0083
Barium (dissolved)	mg/L	0.00326	Antimony (dissolved)	mg/L	0.0054
Beryllium (total)	mg/L	0.000024	Selenium (total)	mg/L	0.00147
Beryllium (dissolved)	mg/L	0.000014	Selenium (dissolved)	mg/L	0.00139
Boron (total)	mg/L	0.121	Tin (total)	mg/L	<0.00001
Boron (dissolved)	mg/L	0.098	Tin (dissolved)	mg/L	0.00025
Bismuth (total)	mg/L	0.000044	Strontium (total)	mg/L	0.0666
Bismuth (dissolved)	mg/L	0.000025	Strontium (dissolved)	mg/L	0.0643
Calcium (total)	mg/L	21.5	Titanium (total)	mg/L	0.00854
Calcium (dissolved)	mg/L	20.5	Titanium (dissolved)	mg/L	<0.00005
Cadmium (total)	mg/L	0.000105	Thallium (total)	mg/L	0.000038
Cadmium (dissolved)	mg/L	0.000007	Thallium (dissolved)	mg/L	0.000036
Cobalt (total)	mg/L	0.000111	Uranium (total)	mg/L	0.000232
Cobalt (dissolved)	mg/L	0.000093	Uranium (dissolved)	mg/L	0.000150
Chromium (total)	mg/L	0.00027	Vanadium (total)	mg/L	0.00043
Chromium (dissolved)	mg/L	<0.00003	Vanadium (dissolved)	mg/L	0.00002
Copper (total)	mg/L	0.00181	Zinc (total)	mg/L	0.012
Copper (dissolved)	mg/L	<0.00002	Zinc (dissolved)	mg/L	0.006

Note: F24 is a combination of several monthly composite samples.

The bulk gravity/flotation test was completed using the same standard conditions applied to the other monthly composite tests. The overall gold recovery (gravity + flotation) was 96.6%, which was comparable to previous years' test results. The rougher tailings were allowed to settle, and the decanted solution was collected and submitted for analysis. A summary is provided in Table 13-13.

Table 13-13: Bulk Testwork Results

Sample	Test Number	Test Size (kg)	P_{80} µm (Rougher Tailings)	Reagents	Product	Weight %	Assays		% Distribution		
									Au		
							Au g/t	S %	Flot (Unit)	Grav + Flot	S
Bulk Composite	F-24 (G-22)	37	70	65 g/t PAX 20 g/t A407 25 g/t MIBC	Mozley Conc. Rougher Conc. Rougher Tailings	 3.8 96.2	 49.1 0.24	 17.5 0.07	 89.0 11.0	69.2 27.4 3.4	 90.8 9.2
					Head (calc.)	100.0	2.09	0.73	100.0	100.0	100.0

Note: Conc. = concentrate; Flot = flotation; Grav = gravity; MIBC = Methyl Isobutyl Carbinol (frother agent); PAX = Potassium Amyl Xanthate (collector agent); A407 = Aero 407 (promotor agent).

13.1.7 2019 Testwork Report Summary

SGS in Lakefield, Ontario, received samples for metallurgical testwork from Harte Gold's operating process plant in late October 2018 and contained in the SGS report (Crary, 2019, March 15). The main objective of their testwork program was to determine the response of the four new variability samples to gravity separation, rougher flotation, and cyanide leaching. Alongside the variability testwork, two flotation concentrate plant samples were received for select tests that investigated cleaner flotation, cyanide leaching, cyanide destruction, and solid/liquid separation testwork. A summary of the head assays is presented in Table 13-14.

Table 13-14: Head Analysis Summary

Element	Unit	Lower Zone South Lens	Lower Zone North Lens	Middle Zone Deep	Upper Zone	Plant Sample Rougher Flotation Concentrate	
						Dry	Wet
Au 1	g/t	10.1	7.28	7.19	5.77	94.0	80.7
Au 2	g/t	8.64	7.22	7.39	6.29	88.2	79.7
Au Avg.	**g/t**	**9.37**	**7.25**	**7.29**	**6.03**	**91.1**	**80.2**
Au Calc.	**g/t**	**5.54**	**6.90**	**3.81**	**3.75**	**76.1**	**78.6**
Ag	g/t	1.3	1.5	1.7	0.8	24.2	33
Cu	%	0.013	<0.01	<0.01	0.014	0.44	0.34
Fe	%	3.62	4.92	5.93	4.66	36.9	38.0
S	%	0.51	0.62	1.53	0.67	21.6	22.2
$S^{=}$	%	0.49	0.60	1.39	0.63	21.1	21.2
SG		-	-	-	-	3.73	3.77
P_{80}	µm	-	-	-	-	67	55

Note: Au Calc. = calculated head from testwork;

The gravity gold recovery values for the variability samples were high, and ranged from 50.3% to 81.3%, averaging 69.4%. The average gold and sulphur flotation recoveries were about 82% and 90%,

at an average mass pull of 3.1%. The average rougher tailing assays were 0.26 g/t Au and 0.12% S. The overall gravity + flotation gold recoveries were high, and averaged about 95%.

The variability and plant flotation concentrates responded well to cyanidation, and after 48 h of leaching the gold extractions were about 96% to 99%. Regrinding the concentrate increased the overall gold extraction and also increased the cyanide and lime consumption. The overall (gravity/flotation/cyanidation) gold recoveries variability samples were 94.0% (without regrinding the flotation concentration) and 94.4% (with regrinding).

In general, the results from the variability samples were comparable to the previous Sugar and Middle Zone test results. The gravity gold recoveries averaged approximately 69%, which compared well to previous results that averaged about 60% to 70%. The overall gravity + flotation test results (~95% average) were in line with past test results, which were about 92% to 97%. The flotation concentrate cyanidation test results compared well to the Sugar Zone results; gold extractions for all tests have been in the 96% to 99% range.

Cyanide destruction testwork was conducted using a leached flotation concentrate plant sample to evaluate the use of SO_2/air for oxidation of cyanide and detoxification of the cyanidation pulp. The objective of the testwork was to produce treated effluent containing <1 mg/L total cyanide.

A series of tests was conducted, and the results indicated that treating the pulp at pH 8.5, at a retention time of about 90 min, with about 7.4 g equivalent SO_2 per gram weak acid dissociable cyanide (CN_{WAD}) (in the feed), produced the desired CND tailing product.

Pressure and vacuum filtration testwork was conducted using the wet flotation concentrate plant sample. One pressure filtration test was completed to assist with the operation of the flotation concentrate filter being used at the site.

13.1.8 Metallurgical Testwork

The main objective of the metallurgical test program was to evaluate the response of the variability samples to a gravity/flotation/cyanidation flowsheet. The as-received flotation concentrate samples were also used for cleaner flotation, cyanidation, and cyanide destruction testwork. The results are presented in the following subsections.

13.1.8.1 Sample Receipt and Preparation

Four variability samples were received at the SGS Lakefield site on 25 October 2018. The sample details are presented in Table 13-15.

Two flotation concentrate plant samples were also received (dry and wet). The samples were used as required for head assays and metallurgical testwork.

The as-received sample P_{80}s were 67 µm (dry) and 55 µm (wet).

Table 13-15: Variability Sample Details

Composite	Hole ID	From	To	Tag Number	Weight (kg)
Lower Zone South Lens	SZ-17-138W	648.8	651.6	787459	2.15
	SZ-17-223W2	950.2	953.2	787460	2.74
				Total	**4.89**
Lower Zone North Lens	SZ-17-216	745.4	748.1	787461	2.67
	WZ-17-75W	780.3	782.8	787462	1.78
	SZ-18-247W2	882.9	885.3	787463	2.20
				Total	**6.65**
Middle Zone Deep	WZ-17-107W3	910.2	914.6	787454	4.40
	WZ-17-75W	885.5	896.1	787455	12.1
				Total	**16.5**
Upper Zone	SZ-17-127	85.3	88.5	787456	3.80
	SZ-17-128	122.3	124.9	787457	2.69
	SZ-17-134	117.5	120.2	787458	2.60
				Total	**9.09**

The complete head analysis for the variability and plant samples is presented in Table 13-16.

Variability in the duplicate direct gold head assays, and disagreement with the calculated gold grades indicated that the samples contain coarse (free) gold. The variability sample's calculated gold head grades ranged from 3.75 to 6.90 g/t Au; silver grades ranged from 0.8 to 1.7 g/t Ag; sulphur grades were 0.51% to 0.67% for three of the four samples. The Middle Zone deep sample contained 1.53% Sulphur.

The calculated gold head grades for the plant flotation concentrate samples were 76.1 and 78.6 g/t Au.

Table 13-16: Head Assays

Element	Unit	Lower Zone South Lens	Lower Zone North Lens	Middle Zone Deep	Upper Zone	Plant Sample Rougher Flotation Concentrate	
						Dry	Wet
Au 1	g/t	10.1	7.28	7.19	5.77	94.0	80.7
Au 2	g/t	8.64	7.22	7.39	6.29	88.2	79.7
Au Avg.	**g/t**	**9.37**	**7.25**	**7.29**	**6.03**	**91.1**	**80.2**
Au Calc.	**g/t**	**5.54**	**6.90**	**3.81**	**3.75**	**76.1**	**78.6**
Ag	g/t	1.3	1.5	1.7	0.8	24.2	33
Cu	%	0.013	<0.01	<0.01	0.014	-	-
Fe	%	3.62	4.92	5.93	4.66	-	-
S	%	0.51	0.62	1.53	0.67	21.6	22.2
$S^{=}$	%	0.49	0.60	1.39	0.63	21.1	21.2
SG		-	-	-	-	3.73	3.77
ICP Scan							
Al	g/t	-	-	-	-	32,100	31,500
As	g/t	-	-	-	-	<30	<30
Ba	g/t	-	-	-	-	123	144
Be	g/t	-	-	-	-	<0.4	0.44
Bi	g/t	-	-	-	-	<20	<20
Ca	g/t	-	-	-	-	31,500	27,600
Cd	g/t	-	-	-	-	66	74
Co	g/t	-	-	-	-	413	379
Cr	g/t	-	-	-	-	1,560	1,580
Cu	g/t	-	-	-	-	4,370	3,420
Fe	g/t	-	-	-	-	369,000	380,000
K	g/t	-	-	-	-	5,860	7,560
Li	g/t	-	-	-	-	<6	<7
Mg	g/t	-	-	-	-	10,600	9,520
Mn	g/t	-	-	-	-	592	560
Mo	g/t	-	-	-	-	73	<40
Na	g/t	-	-	-	-	6,010	5,860
Ni	g/t	-	-	-	-	1,040	1,010
P	g/t	-	-	-	-	786	<300
Pb	g/t	-	-	-	-	4,790	4.580
Sb	g/t	-	-	-	-	<20	<20
Se	g/t	-	-	-	-	<30	<80
Sn	g/t	-	-	-	-	51	<30
Sr	g/t	-	-	-	-	79.8	80.1
Ti	g/t	-	-	-	-	2,050	1,890
Tl	g/t	-	-	-	-	<30	<30
U	g/t	-	-	-	-	<50	<50
V	g/t	-	-	-	-	94	83
Y	g/t	-	-	-	-	7.3	6.7
Zn	g/t	-	-	-	-	6,480	7,290

Note: Au Calc. = calculated head from testwork

13.1.8.2 Gravity and Flotation Testwork (Variability Samples)

In total, eight 2 kg gravity separation and flotation tests were completed, which evaluated the response of the samples to the optimized conditions established during the previous test program (Crary, 2017). The test details and results are presented below.

The gravity and flotation test results from both the kinetic and concentrate generation tests are summarized in Table 13-17.

Table 13-17: Flotation Test Results—Variability Samples

Sample	Product	Weight %	Assays		Distribution %			
					Au			
			Au g/t	S %	Flot (Unit)	Grav	Grav+Flot	S
LZ South	Rougher Conc. 1-5	2.3	50.7	-	81.8	75.9	95.6	-
LZ North	Rougher Conc. 1-5	2.5	47.8	23.3	82.8	80.6	96.7	89.3
MZ Deep	Rougher Conc. 1-5	4.9	26.6	28.3	76.7	50.9	88.6	93.6
Upper Zone	Rougher Conc. 1-5	2.8	33.8	23.7	83.9	67.6	94.8	87.1
LZ South	Rougher Conc. 1-5	2.1	110.0	22.7	90.1	50.3	95.1	90.7
LZ North	Rougher Conc. 1-5	2.3	43.5	23.5	82.4	81.3	96.7	88.7
MZ Deep	Rougher Conc. 1-5	4.8	19.0	29.8	73.6	70.6	92.3	93.1
Upper Zone	Rougher Conc. 1-5	3	24.7	21.4	82.9	78.3	96.3	90.3

Note: LZ = Lower Zone; MZ = Middle Zone.

The gravity gold recovery values for the eight tests ranged from 50.3% to 81.3% and averaged 69.4%. The calculated gold head grades did not compare well to the direct heads, and verified the presence of high levels of coarse gold in these samples.

The average gold and sulphur flotation recoveries (Figure 13-3) were about 82% and about 90% respectively (F-1 not included in sulphur average), at an average mass pull of 3.1%. The average rougher tailing assays were 0.26 g/t Au and 0.12% S. The overall gravity + flotation gold recoveries were high and averaged about 95% over eight tests.

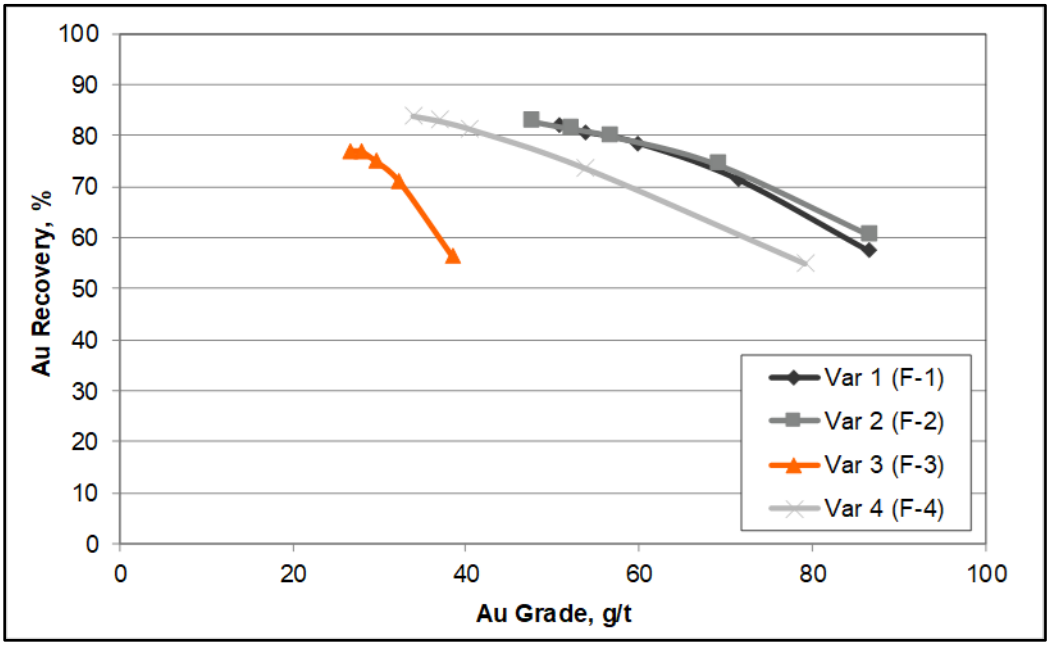


Figure 13-3: Gold Recovery vs. Gold Grade

13.1.8.3 Cleaner Flotation Testwork

One cleaner flotation test was completed using the 23 November 2018 (wet) plant sample. Three cleaner stages (1, 2, and 3 min) were completed and the sample from each stage was assayed for gold and sulphur. The sample was ground for two minutes prior to testing to freshen the sample. The objective of the test was to determine the response of the sample while maintaining a high gold grade and low mass pull. The test results are summarized in Table 13-18.

Table 13-18: Cleaner Flotation Test Results

Sample	Test No.	Reagents	Product	Weight %	Assays		% Distribution		
							Au		
					Au g/t	S %	Flot (Unit)	Grav+Flot	S
Nov 23 2018 Wet Plant Sample	F-5	40 g/t PAX	Cleaner 1	13.4	303	36.5	51.2	-	22.4
		0 g/t 407	Cleaner 2	40.8	173	35.7	88.6	-	66.6
		15 g/t MIBC	Cleaner 3	54.3	143	35.4	97.4	-	87.8
			Head (calculated	100.0	79.6	21.9	100.0	-	100.0
			Head (direct)	-	80.2	22.2	-	-	-

The first cleaner concentrate gold grade and recovery was 303 g/t Au and 51.2% (13.4% mass pull). The gold grade decreased to 143 g/t Au as the mass pull increased to 54.3%. The gold recovery was 97.4% after three stages (6 min flotation). It is important to note that the mass pulls shown are for a 125 g feed sample. The final cleaner mass pull (54.3%) would represent a mass pull of about 3.4% if this was a 2 kg feed charge that produced 125 g of rougher flotation concentrate (6.3% rougher mass pull).

13.1.8.4 Cyanidation Testwork

Four flotation concentrate cyanidation tests were completed using as-received plant samples, and two variability sample tests were completed using the combined concentrate generated from tests F-6 to F-9. The objective of the tests was to determine the impact of regrinding, and to establish the gold extraction (and overall gold recovery) for a gravity + flotation + cyanidation flowsheet. A laboratory pebble mill was used to regrind the flotation concentrate prior to leaching. The cyanidation test conditions were as follows:

Pulp Density = ~20% solids (w/w)
Pulp pH = 10.5–11 maintained with lime
Cyanide Concentration = 1 g/L NaCN maintained
Retention Time = 48 h (subsampled at 6 and 24 h)
Dissolved Oxygen Concentration = ~20–25 mg/L
Pre-aeration = 4 h with oxygen (CN-5 and CN-6 only)

The tests were conducted in standard bottle rolls, and subsamples of the leach solution were taken at specific time intervals and assayed for gold to monitor the rate of extraction. Upon completion of the test, the pulps were filtered, and the final leach solutions were collected and assayed for gold. The filter cakes were washed with water, dried, then assayed for gold (duplicate 30 g cuts by fire assay). Subsamples of the final residue were also submitted for particle-size analysis. Oxygen was sparged into the bottles to main the dissolved oxygen (DO) concentration target. The variability samples include a pre-aeration stage before cyanidation to assess the impact on solution chemistry.

The results are summarized in Table 13-19.

Table 13-19: Flotation Concentrate Cyanidation Results

Sample	CN Test No.	Feed Size P_{80} µm	Reagent Cons. kg/t of CN Feed		Au Extraction (%)			Au Residue (g/t)			Au Head (g/t)	
			NaCN	CaO	6 h	24 h	48 h	A	B	Avg.	Calc.	Direct
Wet Sample—23 Nov 2018	1	55	4.31	0.95	48	90	96.1	3.27	3.05	3.16	80.7	80.2
	2	26	5.88	1.13	66	98	99.2	0.70	0.57	0.64	75.5	
Dry Ro Conc. Sample	3	67	3.86	0.95	44	86	97.3	2.43	2.49	2.46	92.4	91.1
	4	25	6.81	1.37	79	99	99.5	0.45	0.24	0.35	68.3	
Variability Samples Combined Concentrate[1]	5	51	2.33	1.82	88	94	97.3	0.86	0.83	0.85	31.6	40.9[2]
	6	about 15–20	3.87	3.25	94	97	98.8	0.37	0.30	0.34	28.2	

Notes: [1] Combined concentrate from tests F-6 to F-9 (4 variability samples).
[2] Weighted average of F-6 to F-9 concentrate assays.

The flotation concentrates responded well to cyanidation, and after 48 h of leaching the gold extractions were about 96% to 99%. Regrinding the concentrate increased the overall gold extraction and also increased the cyanide and lime consumption. The leach kinetics for each test are presented in Figure 13-4. The results indicated that a small amount of leaching was still occurring between 24 and 48 h for five of the six tests.

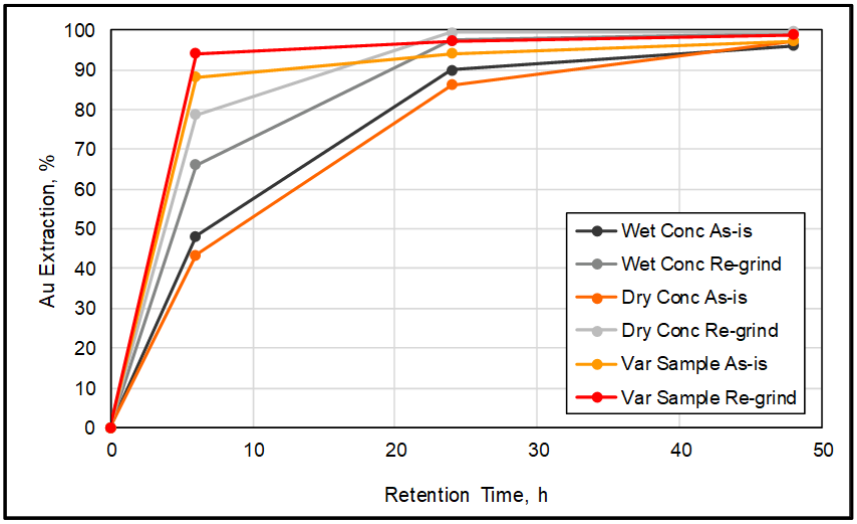


Figure 13-4: Flotation Concentrate Leach Kinetics

13.1.8.5 Bulk Cyanidation Test

A single bulk cyanidation test was completed using the dry plant flotation concentrate sample so that leached pulp could be produced for cyanide detox testwork. Approximately 6.6 kg of flotation concentrate was reground prior to leaching. The pulp density for the test was 40% solids (w/w), and the remaining test conditions were identical to those outlined in Section 13.2.2.3. A 4 h pre-aeration stage (using oxygen) was included prior to leaching.

Upon completing the cyanidation test, and prior to being forwarded for cyanide detox testing, the leached pulp was contacted with carbon for 6 h to simulate a recovery circuit (i.e., remove gold and silver from solution). Table 13-20 summarizes the bulk testwork results.

Table 13-20: Bulk Cyanidation Testwork Summary

CN Test No.	Feed Size P_{80} µm	Reagent Consumption kg/t of CN Feed		Au Extraction (%)			Au Residue (g/t)			Au Head (g/t)	
		NaCN	CaO	6 h	24 h	48 h	A	B	Avg.	Calc.	Direct
7	18	3.71	1.48	-	-	**97.8**	1.80	1.22	1.51	67.8	91.1

The bulk cyanidation gold extraction was about 98%. The extraction was calculated using the barren leach solution, carbon, and final residue grade. The barren leach solution was submitted for cyanide detox feed assays so the results could be used to design the cyanide detox testwork. The solution analysis is presented in Table 13-21.

Table 13-21: CND Feed Assays

Sample	Analysis (mg/L)									
	Au	Ag	Cu	Fe	Zn	Ni	CN_T	CN_{WAD}	CNS	CNO
CN-7 BLS	1.65	2.59	503	7.84	80	7.7	911	824	1,500	17

13.1.8.6 Comparison to Sugar and Middle Zone Results

In general, the results from the variability samples were comparable to the Sugar Zone and Middle Zone test results (Crary, 2019, March 15). The gravity gold recoveries averaged about 69%, which compared well to previous results that averaged about 60% to 70%. The overall gravity + flotation test results (~95% average) were in line with past test results which were about 92% to 97%.

The mass pulls for the variability samples averaged about 3%, which was similar to the Sugar Zone samples, which had values in about the 3% to 4% range. At a mass pull of about 3% to 4%, the Middle Zone overall gold recoveries were about 91% to 92%.

The flotation concentrate cyanidation test results compared well to the Sugar Zone results; gold extractions for all tests have been in about the 96% 99% range. The Middle Zone samples were not submitted for flotation concentrate leach tests.

13.1.8.7 Cyanide Destruction Testwork

Cyanide destruction testwork was completed using the bulk leached sample (CN-7). The objective was to investigate the response and determine if a treated pulp containing <1 mg/L total cyanide (CN_T) was possible using the SO_2/air detoxification process. Detailed test conditions and results are shown in Table 13-22.

The tests were conducted at 40% solids (w/w), and the pH target was 8.5. All tests were conducted at room temperature. The first continuous test (CND 1-1) was stopped after 80 min (test duration). The CN_{WAD} (by picric) was 86.2 mg/L using an addition of 5.49 g equivalent SO_2 per gram of CN_{WAD} in the feed. The SO_2 addition was increased to 7.08 g/g in the second test (CND 1-2). The retention time remained at about 60 min, but after running for 45 min (test duration), it was clear that this test was also not going to achieve the objective. Therefore, a third test was completed with the SO_2 addition increased slightly to 7.37 g/g and the test retention time increased to 94 min. The test was successful, and the CN_{WAD} (by picric) decreased quickly as shown in Figure 13-5. The final solution contained 0.40 mg/L CN_T, and <1 mg/L CN_{WAD} (confirmed by analytical method and picric analysis).

Table 13-22: Cyanide Destruction Test Results—SB Master Composite

			Product (Solution Phase)								Reagent Addition								
					CNWAD by						CN_{WAD} (g/g)			Feed Pulp (g/L)			Solids (kg/t)		
Test	Test Duration (min)	Retention Time (min)	pH	CN_T (mg/L)	Analytical Method Lab (mg/L)	Picric Acid (mg/L)	CNO	CNS	Cu (mg/L)	Fe (mg/L)	SO_2 Equiv.	Lime	Cu	SO_2 Equiv.	Lime	Cu	SO_2 Equiv.	Lime	Cu
Feed (CN-7	-	-	*10.6*	*911*	*824*	-	*17*	*1,500*	*503*	*7.84*	-	-	-	-	-	-	-	-	-
Batch Test																			
1	390	390	8.5	-	-	2.67	-	-	-	-	22.7	9.50	0	15.1	6.38	0	28.0	11.7	0
Continuous Tests																			
1-1	80	66	8.4	-	-	86.2	-	-	-	-	5.49	4.98	0	3.66	3.40	0	6.79	6.16	0
1-2	45	61	8.5	-	-	76.1	-	-	-	-	7.08	3.97	0	4.71	2.71	0	8.76	4.91	0
1-3	210	94	8.5	0.40	<0.1	0.99	1,300	970	7.5	<0.1	7.37	5.27	0	4.88	3.59	0	9.11	6.52	0

Note: "-" = no sample submitted for assays.

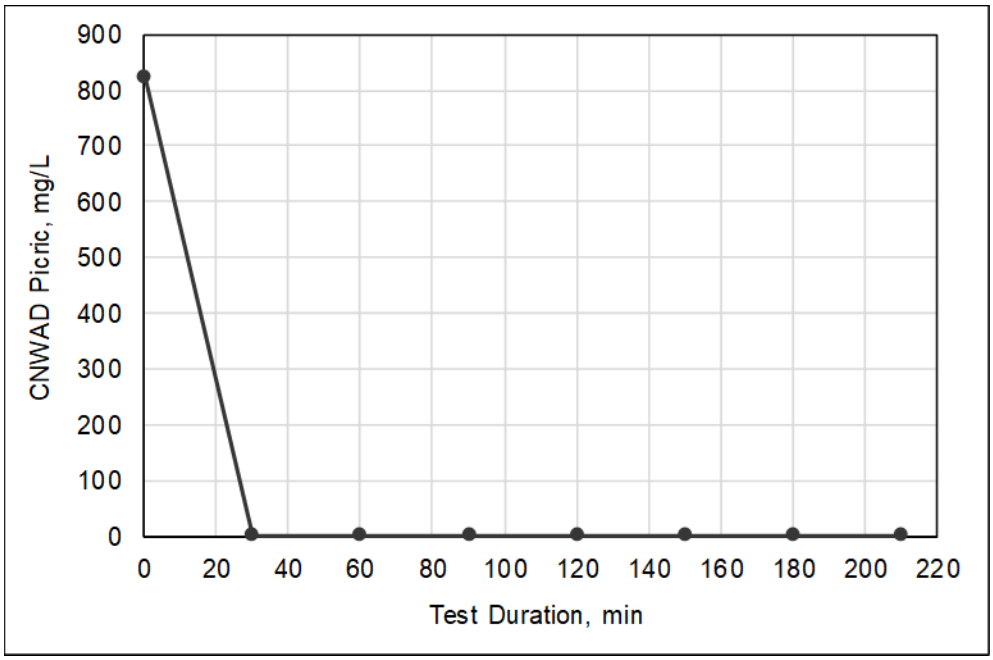


Figure 13-5: CN_{WAD} (Picric) vs. Test Duration

13.1.8.8 Solid/Liquid Separation Testwork

Pressure and vacuum filtration tests were conducted on a sample identified as "Plant Sample 23 November 2018." The as-received plant sample was subjected to pressure and vacuum filtration and a reground plant sample was subjected to vacuum filtration testing as a simulated CND discharge stream. Cake washing was also conducted on the reground plant sample after the vacuum filtration test was completed.

To differentiate the samples, the reground plant sample was identified as "Plant Sample Reground."

13.1.8.9 Pressure Filtration

A pressure filtration scoping test was conducted using conditions specified by the client. The conditions were as follows: 60% w/w solids filter feed; 5.5 bar pressure level; 16 mm cake thickness; no air drying. The intent of the scoping test was to determine if the sample could be dewatered to less than 10% w/w residual moisture using a filter aid in the filter feed. SGS suggested using a coagulant, namely BASF Magnafloc 1687. In total, three filtration tests were performed on the as-received plant sample. All tests were conducted under the same conditions, except for the coagulant dosage. One test was performed as a baseline, without coagulant addition. Two tests were performed using a coagulant addition to the filter feed, one test at 50 g/t Au and the other at 100 g/t Au.

The tests conducted with coagulant as filter aid indicated that higher throughput could be achieved when compared to the test conducted without coagulant addition; however, the residual cake moisture was similar for all tests, at an average of 10.1% w/w moisture. The surfaces of all filter cakes were dry to the touch. The results are summarized in Table 13-23.

Table 13-23: Pressure Filtration Scoping Results Summary—Plant Sample 23 Nov 2018

Sample ID	Filter Cloth	Operating Conditions							Filter Outputs				
		d_{80} µm	Feed Solids (% w/w)	Filter Aid[1] (g/t)	Pressure Level (bar)	Form Time (s)	Dry Time (s)	Form/Dry Ratio	Cake Thickness[2] (mm)	Throughput[3] (dry kg/m²•h)	Cake Moisture (% w/w)	Filtrate TSS	Cake Texture
Plant Sample 23 Nov 2018	Testori P4408 TC	57	60.0	None	5.5	38	67	0.56	16.0	1,250	10.3	79	DTT
				50		25	58	0.52	16.0	1,811	10.0	94	DTT
				100		22	42	0.51	16.0	2,086	10.1	92	DTT

Notes: [1] BASF Magnafloc 1687, a diallyldimethylammonium chloride homopolymer coagulant.
[2] Represents half of the chamber thickness.
[3] Calculated using cycle time, which includes form and dry times only.
DTT = dry to the touch.

13.1.8.10 Vacuum Filtration

Vacuum filtration tests were conducted on the as-received Plant Sample 23 November 2018 and the Plant Sample Reground at 60% w/w solids in the filter feed. Based on the results of the pressure filtration scoping tests, 100 g/t BASF Magnafloc 1687 was added to the filter feed as a filter aid. The tests were conducted at 21 inches of mercury (0.71 bar) vacuum level. The results are summarized in Table 13-24 and Table 13-25.

Table 13-24: Vacuum Filtration Results Summary—Plant Sample 23 Nov 2018

Sample ID	Filter Cloth	Operating Conditions							Filter Outputs				
		d_{80} µm	Feed Solids (% w/w)	Filter Aid[1] (g/t)	Vacuum Level (inch Hg)	Form Time (s)	Dry Time (s)	Form/Dry Ratio	Cake Thickness (mm)	Throughput[2] (dry kg/m²·h)	Cake Moisture (% w/w)	Filtrate TSS	Cake Texture
Plant Sample 23 Nov 2018	Testori P4408 TC	57	60.0	100	21	352	35	10.12	31.0	557	19.5	6	Sticky
						150	30	5.00	20.0	790	19.0	11	Sticky
						150	75	2.00	20.0	632	18.1	10	DTT
						156	156	1.00	20.0	454	17.4	13	DTT
						160	320	0.50	20.0	292	17/3	13	DTT
						51	153	0.33	10.0	337	16.3	31	DTT
						349	210	1.66	30.0	374	18.2	13	Sticky

Notes: [1] BASF Magnafloc 1687, a diallyldimethylammonium chloride homopolymer coagulant.
[2] Calculated using cycle time which includes form and dry times only.
DTT = dry to the touch.

Table 13-25: Vacuum Filtration Results Summary—Plant Sample Reground

Sample ID	Filter Cloth	Operating Conditions							Filter Outputs				
		d_{80} µm	Feed Solids (% w/w)	Filter Aid[1] (g/t)	Vacuum Level (inch Hg)	Form Time (s)	Dry Time (s)	Form/Dry Ratio	Cake Thickness (mm)	Throughput[2] (dry kg/m²·h)	Cake Moisture (% w/w)	Filtrate TSS	Cake Texture
Plant Sample Reground	Testori P4408 TC	26	60.0	100	21	328	33	9.95	32.0	530	24.6	9	Wet
						136	27	50.04	21.0	769	24.2	27	Wet
						131	65	2.02	21.0	652	23.7	21	Wet
						135	135	1.00	20.0	474	23.3	24	Sticky
						137	284	0.50	21.0	310	23.0	24	Sticky
						41	123	0.33	10.0	387	23.0	45	Sticky
						326	183	1.78	32.0	371	22.8	23	Sticky

Notes: [1] BASF Magnafloc 1687, a diallyldimethylammonium chloride homopolymer coagulant.
[2] Calculated using cycle time which includes form and dry times only.

13.1.8.11 Vacuum Filtration—Plant Sample 23 Nov 2018

Cake thicknesses were 10 to 31 mm. The resulting solids output (i.e., dry solids capacity) was 292 to 790 kg/m^2·h. The discharge cake residual moisture content was 16.3% to 19.5% w/w moisture. Filtered cake surface texture ranged from sticky to dry to the touch.

13.1.8.12 Filtration-Cake Washing—Plant Sample Reground

Vacuum filtration-cake washing was conducted on the Plant Sample Reground. To track washing efficiency, the liquid fraction of the sample was spiked with nickel sulphate ($NiSO_4$) to a Ni tenor of 2,000 mg/L. Recovery of the spiked Ni was about 97% after 0.6 w/w wash ratio (versus the total solute in the feed). The recovery reached 100% after 0.9 w/w wash ratio.

13.2 2020 Testwork Summary

13.2.1 Middle Zone Comminution Testwork Results

In October 2020 Harte Gold undertook a ball mill bond work index testwork program of Middle Zone ore. The testwork was completed by Met-Solve and was performed on drill core samples from selected diamond drill holes that intersected the Middle Zone ore body in areas that are expected to be mined over the next four years.

13.2.1.1 Middle Zone Samples

Figure 13-6 shows a long section of the specific drill hole intersects that were chosen for this testwork program.

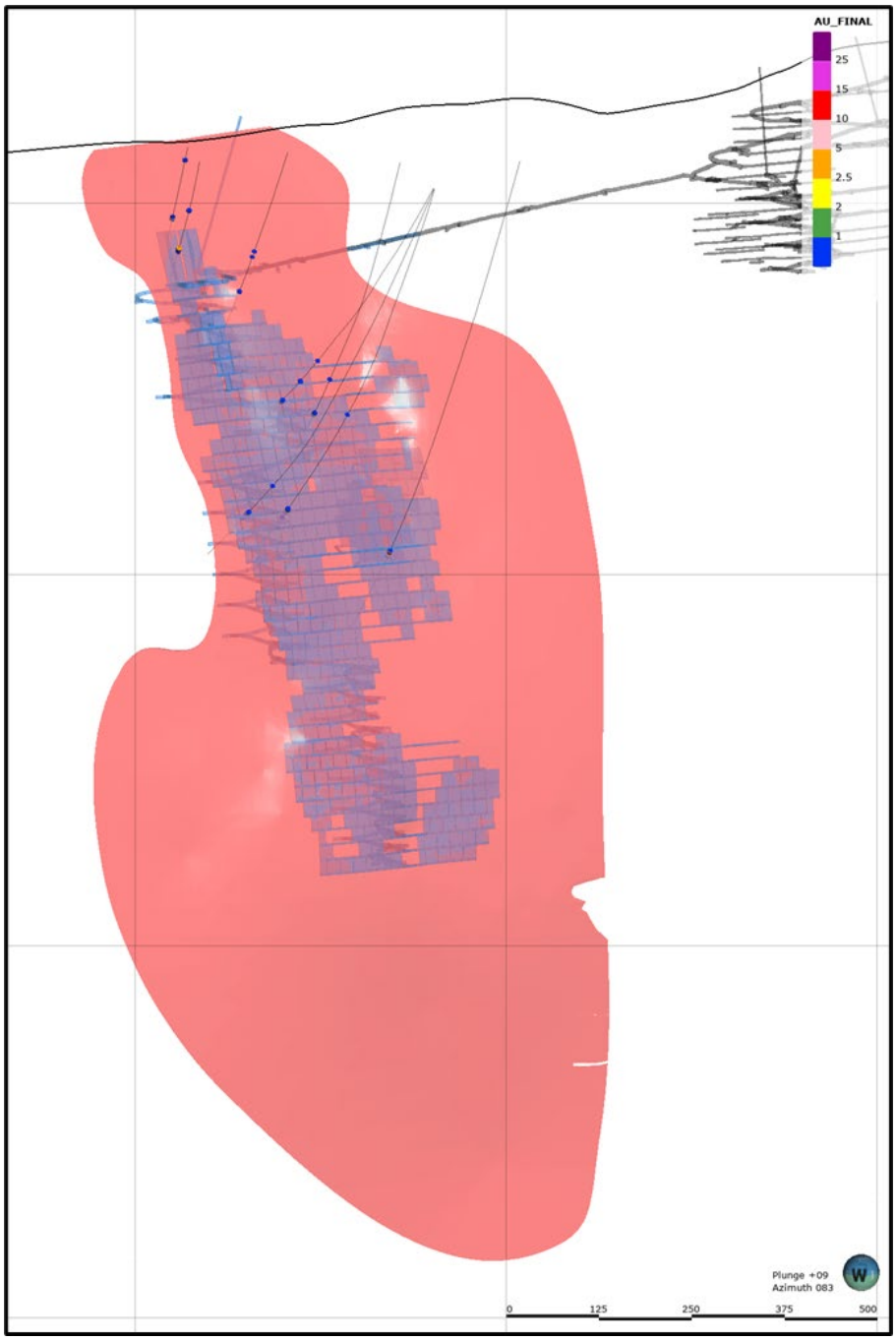


Figure 13-6: Harte Gold Middle Zone Sample Location

13.2.1.2 Middle Zone Bond Mill Grindability

The BWi was calculated and a summary of these results is shown in Table 13-26.

Table 13-26: Harte Gold Middle Zone Grindability Test Results

Measurement	Unit	Sample
Bulk Density	g/cm^3	1.88
Feed P_{80}	µm	2,500
Final Undersize P_{80}	µm	144
Grindability	(g) net undersize/rev	1.64
Bond Work Index	kWh/t	12.7

Note: The bulk density was calculated from the mass of the 700 cm^3 feed subsample.

13.2.2 Sugar Zone Comminution Testwork Results

Harte Gold contracted Met-Solve Laboratories Inc. (Met-Solve) to conduct a series of comminution testwork on the plant's ball mill feed (BMF) and primary jaw crusher product obtained from the Sugar Zone Mine Operation.

The objective of the testwork was to assess the comminution circuits at the Sugar Zone Mine through laboratory testwork, and to provide ore characterization data for a mill expansion evaluation project.

Four samples were received over the duration of the testwork program. A summary of the received samples and their corresponding weights is presented in Table 13-27.

Table 13-27: Harte Gold Samples Received by Met-Solve Laboratories

Sample	Weight (kg)	Date Received
Nov 2019 BMF sample for Gravity Circuit Audit	67.4	Dec 18, 2019
Mar 2020 Ball Mill Feed	6.46	Jul 13, 2020
Aug 16-30, 2020 Ball Mill Feed	127.2	Sep 21, 2020
Aug 16-30, 2020 Jaw Crusher Product	115.2	Sep 21, 2020

13.2.2.1 Scope of Testwork

The various Harte Gold samples were subjected to the following comminution tests. The BWi test was conducted at different closing mesh sizes to straddle the current plant cyclone overflow particle size of 100 µm (P_{80}).

13.2.2.2 November 2019 Ball Mill Feed Sample

Three BWi tests were conducted on the ball mill feed sampled from the plant in November 2019. BWi tests were conducted at different closing mesh sizes—106 µm, 150 µm, and 212 µm—to obtain a particle size range above and below the measured plant cyclone overflow particle size of 100 µm (P_{80}). The BWi test results of the November 2019 ball mill feed sample are summarized in the Table 13-28.

Table 13-28: November 2019 Harte Gold ball mill feed Grindability Results

Test Closing Mesh		Feed (F_{80} µm)	Product (P_{80} µm)	Grams/rev	BWi kWh/t	Feed % pass Undersize	Levin B kWh/rev
Mesh	µm						
70	212	2,275	151	3.07	9.44	22.87	2.27E-05
100	150	2,275	107	2.51	9.61	19.93	2.28E-05
140	106	2,275	77	2.09	9.86	17.08	2.31E-05

The initial BWi tests on the November ball mill feed sample reported an average work index of 9.64 kWh/t. This would classify the sample as having a moderately soft hardness.

13.2.2.3 March 2020 Ball Mill Feed Sample

The November 2019 ball mill feed sample reported work index value was mentioned by the client to be softer than expected when compared to the 14.7 kWh/t work index SGS (Crary, 2017) reported on samples taken in 2017.

The BWi test at 150 µm closing screen size was repeated on the March 2020 ball mill feed grab sample taken for verification.

The site prepared March ball mill feed sample initially contained too great a volume of fines to be suitable for the BWi test. Under the recommendations of Mr. Alex Doll, following discussion with the client, the March ball mill feed composite was stage-crushed and reconstituted to match the starting particle-size distribution of the November 2019 composite prior to conducting the BWi test.

The BWi test result of the March 2020 ball mill feed composite is presented in Table 13-29.

Table 13-29: March 2020 Harte Gold Ball Mill Feed Grindability Results

Test Closing Mesh		Feed F_{80} µm	Product (P_{80} µm)	Grams/rev	BWi (kWh/t)	Feed % pass Undersize	Levin B (kWh/rev)
Mesh	µm						
100	150	2275	107	2.39	10.01	19.94	2.26E-05

The March ball mill feed composite had a slightly higher work index of 10.01 kWh/t compared to the November ball mill feed sample, giving confidence that the mill was processing a softer ore zone.

13.2.2.4 August 2020 Ball Mill Feed Composite Sample

Upon the mill restart and stabilization at Sugar Zone Mine, the ball mill feed stream was sampled over a 2-week period (from 16 to 30 August 2020) to form a composite and representative sample. The purpose of this composite is to provide definitive comminution data for a mill expansion study.

In a similar approach to the November 2019 ball mill feed sample, three BWi tests were carried out at 106 µm, 150 µm, and 212 µm closing mesh sizes to straddle the measured plant cyclone overflow particle size of 100 µm (P_{80}).

The BWi test results of the November 2020 BMF sample are summarized in Table 13-30.

Table 13-30: August 2020 Harte Gold ball mill feed Composite Grindability Results

Test Closing Mesh		Feed (F_{80} µm)	Product (P_{80} µm)	Grams/rev	BWi (kWh/t)	Feed %pass Undersize	Levin B (kWh/rev)
Mesh	µm						
70	212	2,337	142	3.00	9.18	27.23	2.39E-05
100	150	2,337	106	2.54	9.44	23.88	2.40E-05
140	106	2,337	79	1.93	10.66	20.82	2.38E-05

The BWi tests on the August 2020 ball mill feed sample reported an average work index of 9.73 kWh/t.

A total of three Bond ball mill work index tests were repeated on the August 2020 ball mill feed sample at 150 µm closing screen size as a quality check. Subsamples of the BMF were submitted to ALS and Actlabs for sample preparation and BWi testing. Additionally, a separate stage crush and BWi test was repeated internally by Met-Solve. The results of the quality check tests are presented in Table 13-31.

Table 13-31: August 2020 Harte Gold ball mill feed Composite Quality Check of Grindability Results

Lab	Test Closing Mesh		Feed (F_{80} µm)	Product (P_{80} µm)	Grams/rev	BWi (kWh/t)	Feed %pass Undersize	Levin B (kWh/rev)
	Mesh	µm						
ALS QC	100	150	1,943	110	2.11	11.57	27.62	2.45E-05
MSL Repeat	100	150	2,212	102	2.48	9.46	27.33	2.51E-05
Actlabs QC	100	150	2,600	113	2.21	10.86	17.4	2.16E-05

Note : MSL=Met Solve Labs

There was good repeatability of results conducted within Met-Solve with the repeat test reporting a BWi of 9.46 kWh/t compared to the 9.44 kWh/t value obtained in the initial test.

Regarding the discrepancy between Met-Solve and ALS, it was found that the two labs use identical mill equipment and media procured from BICO Braun International, USA. The labs' consensus was that the difference in their results is related to mill wear, and that the results can differ by 1–2 kWh/t based on the lab mill's internal profile. For example, given two identical mills, the older mill should yield a higher work index value as the inner wall wears, becomes smoother, and its diameter increases. Thus, to achieve the same target size an ore material requires more revolutions on the older ALS mill compared to the Met-Solve mill.

These comments were corroborated in email statements made by Mr. Alex Doll during his audit of Met-Solve's 10-year-old BWi mill at the early stages of this campaign.

13.2.2.5 August 2020 Primary Jaw Crusher Sample

The primary jaw crusher product stream was sampled over a 2-week period (from 16 to 30 August 2020) to form a composite upon the Sugar Zone Mine mill restart and stabilization. The composite sample was subjected to a series of CWi, RWi, and comparative BWi tests. A CWi test and RWi test at 1,180 µm closing screen size were conducted by ALS metallurgy.

An average CWi of 11.5 kWh/t was recorded for the jaw crusher sample, which would be characterized as soft. Table 13-32 summarizes the jaw crusher CWi test results. There was some variability of rock hardness present within the sample, from very soft (CWi 2.3–6.6 kWh/t) to hard (CWi 17.9–21.8 kWh/t)

Table 13-32: August 2020 Harte Gold Jaw Crusher Product CWi Results

Crush Work Index	CWi (kWh/t)
Average	**11.48**
Minimum	2.3
Maximum	21.8
Standard Deviation	5.9

An RWi of 13.4 kWh/t was recorded for the Jaw Crusher sample which would be considered moderately soft with respect to rod milling. Table 13-33 summarizes the RWi test.

Table 13-33: August 2020 Harte Gold Jaw Crusher Product RWi Results

Test Closing Mesh		Feed (F_{80} µm)	Product (P_{80} µm)	Grams/rev	RWi (kWh/t)	Feed % pass Undersize
Mesh	µm					
16	1,180	10018	895	10.26	13.37	20.08

The jaw crusher sample was screened at 2" with the +2" and –2" size fraction subjected to separate BWi tests at 150 µm closing size to verify if the –2" fraction was softer, which the client indicated to have a higher gold grade.

The results are presented in Table 13-34. The results show that the finer –2" size fraction of the jaw crusher sample is softer, with a BWi value of 9.33 kWh/t compared to the +2" coarse fraction, which has a value of 9.89 kWh/t.

Table 13-34: August 2020 Harte Gold Jaw Crusher Product BWi Results

Size Fraction	Feed (F_{80} µm)	Product (P_{80} µm)	Grams/rev	BWi (kWh/t)	Feed % pass Undersize	Levin B (kWh/rev)
-2" Undersize	2,492	105	2.53	9.33	20.35	2.30E-05
+2" Oversize	2,444	105	2.36	9.89	19.36	2.24E-05

13.2.3 October 2020 Leaching Testwork Results

One gravity concentrate and one flotation concentrate were received at SGS Lakefield for a metallurgical test program. The objective of the metallurgical program was to evaluate the response of the samples to cyanide leaching.

The gravity concentrate was used for two intensive cyanide leach tests. The flotation concentrate was submitted for a series of tests, and the results were tested to evaluate leaching potential for a carbon-in-pulp (CIP) study. The flotation concentrate was first reground to a grind size P_{80} of 25 µm, then submitted for the following tests:

- Leach Kinetic Test—72 h test with kinetic subsamples to establish the rate of gold dissolution.
- Bulk Leach Test—to provide leached pulp for subsequent carbon contact testing.
- Adsorption Kinetic Test—72 h test with kinetic subsamples to determine the rate at which gold leaves the solution and loads onto the carbon.
- Equilibrium Isotherm Test—a series of bottle roll tests wherein predetermined amounts of pulp were contacted with predetermined amounts of carbon to achieve various target solution/carbon ratios (or targeted carbon loadings).

13.2.3.1 Sample Receipt, Preparation, and Head Analysis

The gravity and flotation concentrate plant samples were received on 28 October 2020 and stored in a refrigerator.

The gravity concentrate sample received was prepared and split into three equal portions, with two being bottled and the third being stored as wet filter cake. Direct head assays were not completed, and the gold grade was determined from the cyanidation tests (calculated head). The particle size analysis (6-400 mesh) of the gravity concentrate (CN-1R residue) is presented in Figure 13-7.

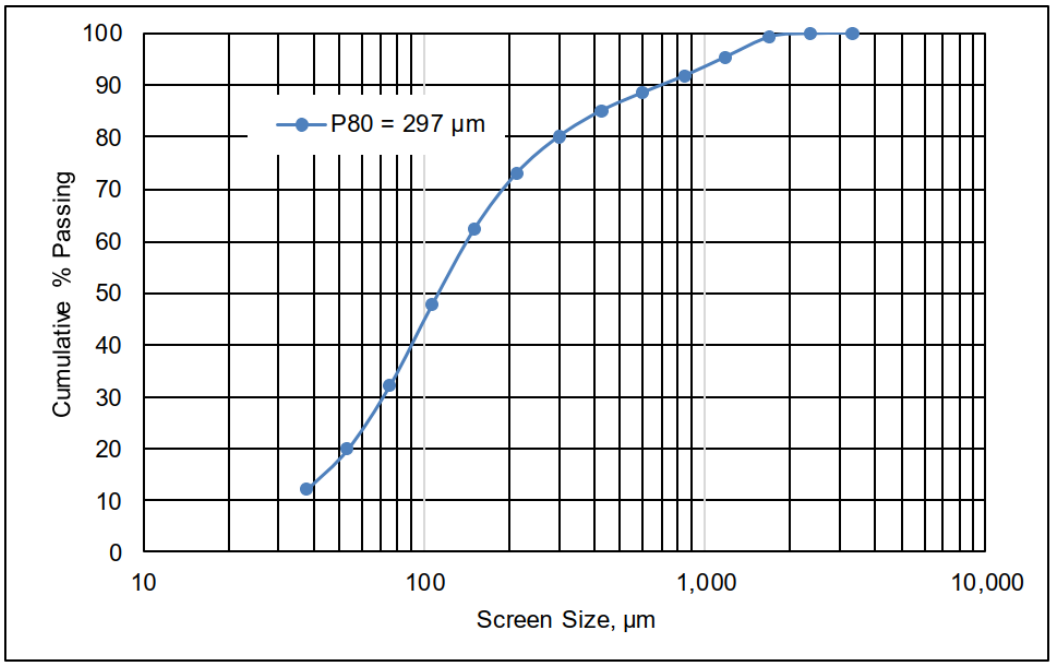


Figure 13-7: Particle Size Analysis—As-Received Gravity Concentrate

The flotation concentrate received was pulped and then passed through a 10-way rotary pulp splitter. The charges (~1.5 kg dry equiv. each) were stored in a refrigerator prior to being reground and used for cyanidation and carbon contact tests. One of the charges was submitted for head assay and a second was used for a grind calibration test. The flotation concentrate head assay results are presented in Table 13-35. The particle size analysis (65–400 mesh) of the as-received sample is presented in Figure 13.8. The flotation concentrate was reground to a grind size P_{80} target of 25 µm prior to cyanidation testwork.

Table 13-35: Head Assays

Element	Flotation Concentrate	Element	Flotation Concentrate
Au g/t Cut 1	109	Li g/t	<20
Au g/t Cut 2	114	Mg g/t	6,600
Au Avg. g/t	112	Mn g/t	489
Au Calc. g/t[1]	114	Mo g/t	97
S %	26.2	Na g/t	4,820
$S^{=}$ %	25.3	Ni g/t	1,180
Ag g/t	63	P g/t	358
Al g/t	22,200	Pb g/t	9,390
As g/t	<40	Sb g/t	<10
Ba g/t	100	Se g/t	<30
Be g/t	0.32	Sn g/t	<20
Bi g/t	42	Sr g/t	60.8
Ca g/t	19,300	Ti g/t	1,410
Cd g/t	122	Tl g/t	<30
Co g/t	450	U g/t	<70
Cr g/t	1,890	V g/t	67
Cu g/t	4,180	Y g/t	4.5
Fe g/t	415,000	Zn g/t	12,700
K g/t	4,660		

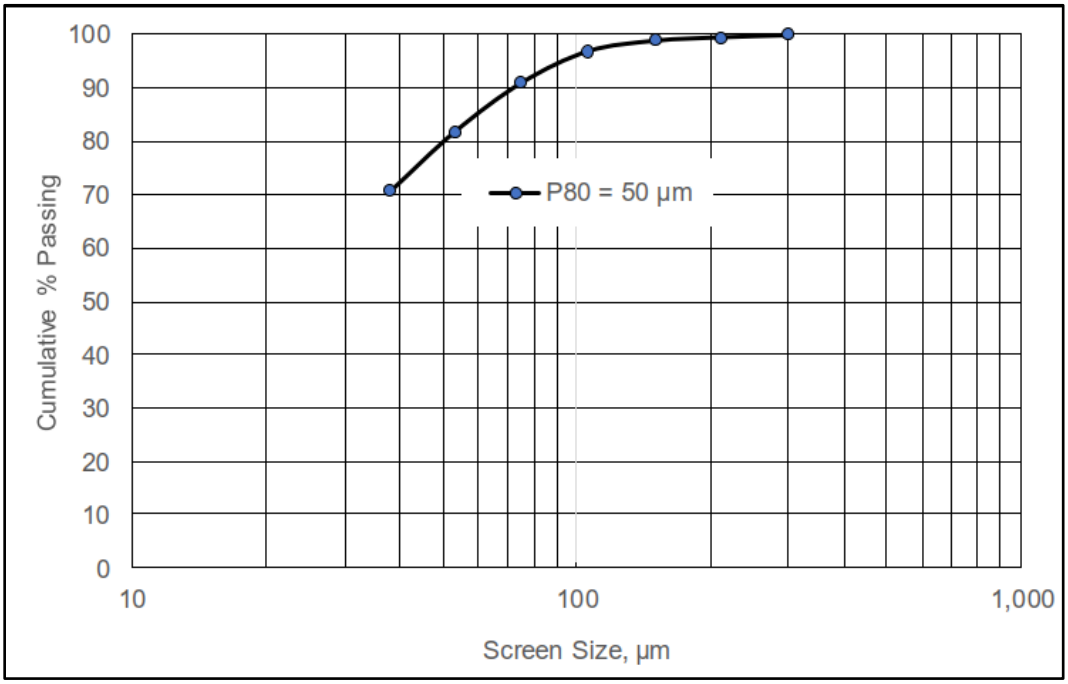


Figure 13-8: Particle Size Analysis—As-received Flotation Concentrate

The flotation concentrate gold head grade was 112 g/t, which compared well to the calculated head grade (114 g/t). The sulphur and sulphide head grades were 26.2% and 25.3%, respectively.

13.2.3.2 Gravity Concentrate Testwork

The gravity concentrate was submitted for cyanidation testwork using standard intensive conditions. The following conditions were used for the tests:

- Bottle roll tests, about 500 g/test (as-received sample)
- 30% solids (w/w)
- Room temperature
- No pre-aeration
- pH 10.5–11 to start test (pH increased with high NaCN addition)
- 20 g/L NaCN maintain
- DO concentration
- CN-1 = no sparging, DO increased naturally to about 5–8 mg/L
- CN-1R = oxygen was sparged, DO target was about 25 mg/L
- 48 h retention time
- LeachAID addition
- CN-1 = 5 g added at start of test
- CN-1R = 20 g added at start of test.

Solution subsamples (10 mL) were taken during the tests to determine the leach kinetics. Upon completion of the tests, the leached pulps were filtered, and the pregnant leach solutions were submitted for gold analysis. The residues were washed, dried, and submitted for duplicate gold assays (30 g—fire assay to extinction). A subsample from CN-1R was submitted for sieve analysis to determine the particle size P_{80} of the as-received sample.

The test results are presented in Table 13-36, and complete test details are appended to this report.

Table 13-36: Cyanidation Test Results

CN Test No.	Feed Size (P_{80} µm)	Reagent Const kg/t of CN Feed		Au Extraction (%)				Au Residue (g/t)			Au Head g/t Calc.	*Normalized Au%[1]*
		NaCN	CaO	6 h	12 h	24 h	48 h	A	C	Avg.		
1	297	36.6	0.00	80.5	81.0	83.6	87.8	695	795	745	6,117	*85.4*
1R	297	23.6	0.00	97.1	98.3	98.6	99.1	38.8	35.9	37.4	4,109	*99.3*
											5,113	

Note: [1] Calculated using the average calculated head (5,113 g/t Au).

The initial test results (CN-1) illustrated that the gravity concentrate needed a higher LeachAID addition and DO concentration. The gold extraction from CN-1 was approximately 88% after 48 h. The gold extraction in test CN-1R increased when the LeachAID addition was increased from 5 to 20 g. The DO concentration was also increased from approximately 8 mg/L to approximately 25 mg/L. The test conditions were more intense for CN-1R, and the gold extraction was approximately 97% after 6 h, then increased to approximately 99% at the end of the test. The results indicated that the gravity concentrate responds well to intensive leach conditions, and high extractions could be achieved if it is directly leached. Additional testwork is recommended to further optimize the reagent additions, but it is likely the gold residue grade of 37 g/t achieved in test CN-1R will be reduced if the gravity leach residue is combined with the float concentrate and reground prior to re-leaching in the float concentrate leach circuit.

The leach kinetic results for the tests are illustrated in Figure 13-9.

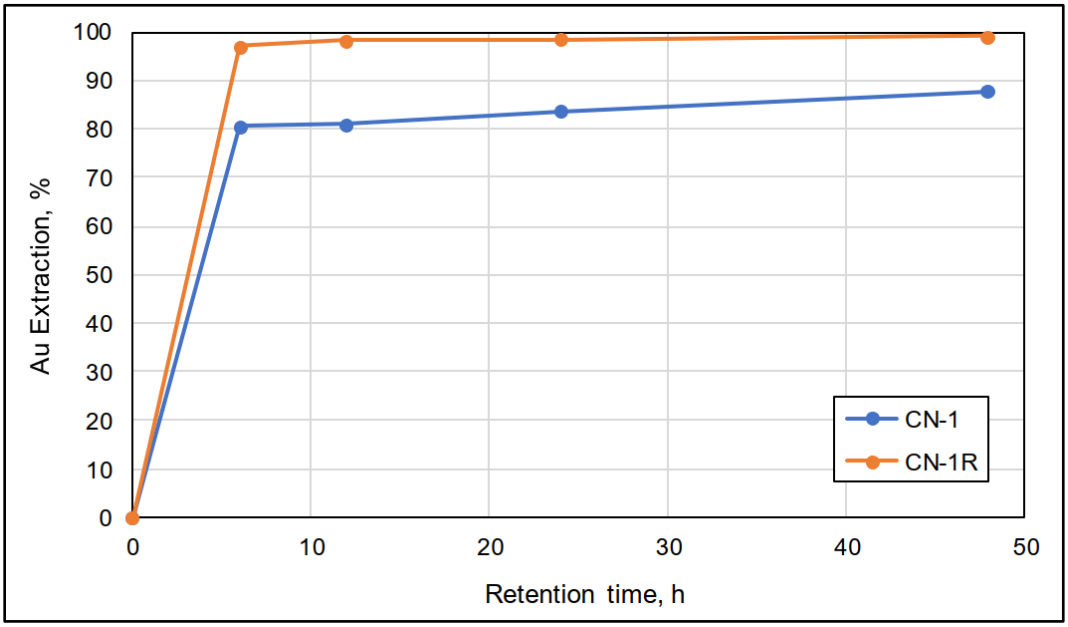


Figure 13-9: Gold Leaching Kinetics

13.2.3.3 Flotation Concentrate Grinding

The flotation concentration was ground to a P_{80} of 27 µm in 500 g batches using a pebble mill before being used for cyanidation testwork and subsequent carbon contact tests. The grind size was confirmed with a Malvern Mastersizer. The particle size distribution is shown in Figure 13-10.

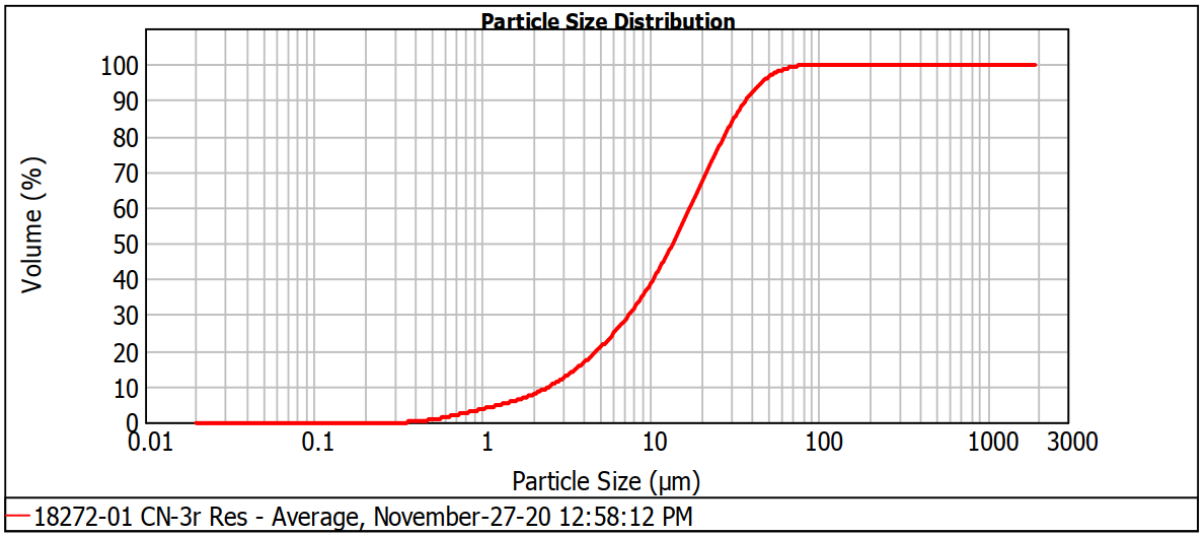


Figure 13-10: Grounded Flotation Concentrate Particle-Size Distribution

The objective of the flotation concentrate testwork was to determine the samples' amenability to cyanide leaching using previous testwork conditions, and to generate leached pulped that could be used for carbon contact testwork. The test results are presented in the subsections below.

13.2.3.4 Flotation Concentrate Leach Kinetics

The leach kinetic test (CN-3R) was conducted in a rolling bottle using the following conditions:

- 35% solids (w/w)
- Room temperature
- Pre-aeration = 4 h
- pH 10.5–11 maintained with lime
- 1 g/L NaCN maintained
- DO Concentration of approximately 20–25 mg/L (oxygen sparged into bottle)
- 72 h retention time.

Solution subsamples (10 mL for Au) were taken during the test at 1, 2, 4, 8, 12, 24, and 48 h to ensure there were sufficient data to determine the leach kinetics. Upon completion of the test, the pulp was filtered, and the final pregnant leach solution (PLS) was submitted for gold analysis. The residue was washed, dried, and submitted for gold assay. The results are presented in Table 13-37 and Figure 13-11. The final bulk leach pregnant leach solution contained 55.5 mg/L Au, similar to the leach kinetic bottle (59.8 mg/L). The kinetic profile indicates gold recovery may have improved with a longer retention time. The CN-4RL pulp was used for isotherm tests as presented in Section 13.3.3.6.

Table 13-37: Flotation Concentrate Leach Results

Product	Au Assays (mg/L)	Au Distribution (%)
1 h PLS	11.8	18.4
2 h PLS	15.6	24.3
4 h PLS	19.6	30.4
8 h PLS	23.5	36.5
12 h PLS	33.8	52.4
48 h PLS	53.2	81.9
72 h PLS	59.8	92.0
Final Residue	8.74	8.0
Head (calc.)	110	100.0
Head (dir.)	112	

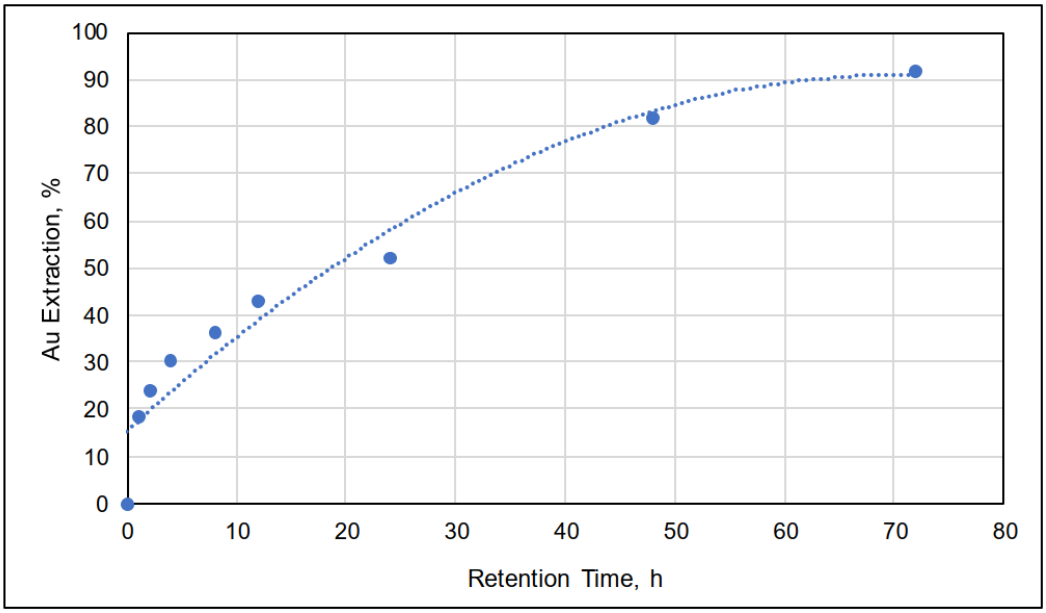


Figure 13-11: Flotation Concentrate Leach Kinetics

13.2.3.5 Flotation Concentrate Carbon Adsorption Kinetics

A second bottle roll test (CN-5) was completed to create sample for the adsorption kinetic test (CN-7). Test CN-5 was a duplicate bottle that ran alongside the leach kinetic test (CN-3R). The pulp from test CN-5 was contacted with carbon (fresh attritioned Calgon GRC-22 carbon) in a batch test in a rolling bottle for 72 h. Solution samples were removed for gold analysis at specific time intervals (1, 2, 4, 8, 24, 48, and 72 h) to establish the adsorption kinetic profile. The slurry was screened at the end of the test to recover the carbon for assay, and the final solution and washed solids were also analyzed for gold to establish a metal balance. The test results are presented in Table 13-38 and Figure 13-12.

Table 13-38: Carbon Adsorption Kinetics Summary

Time (h)	Au in Solution (mg/L)
1	82.0
2	49.9
4	35.9
8	23.6
12	16.0
24	7.52
48	5.33
72	5.34

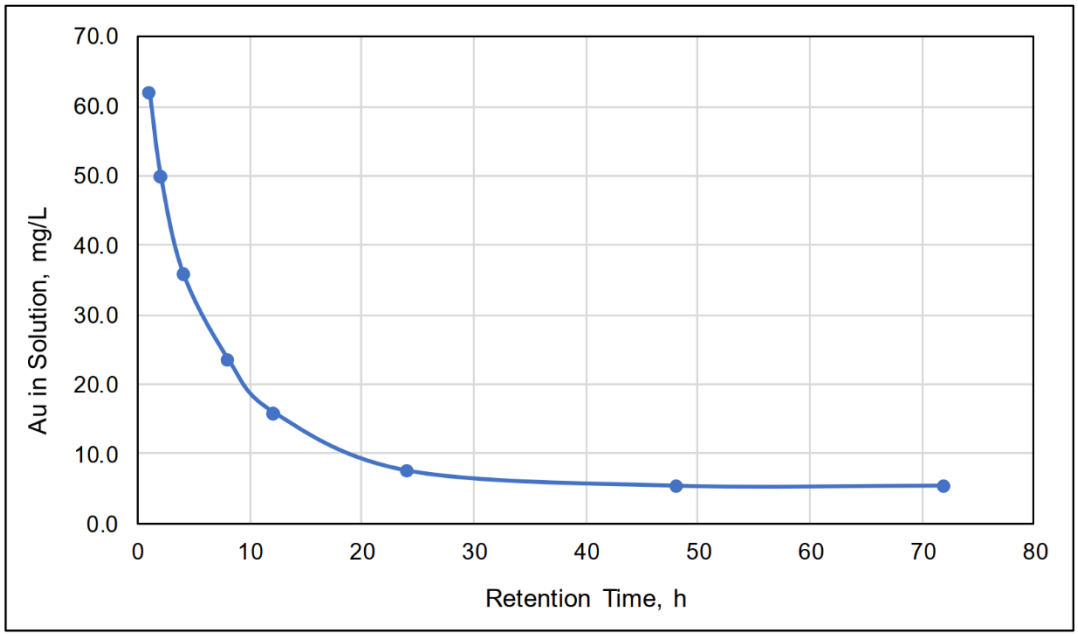


Figure 13-12: Kinetics of Gold Cyanide Extraction by Carbon

13.2.3.6 Flotation Concentrate Carbon Adsorption Equilibrium Isotherms

To generate data for the pseudo equilibrium isotherm, predetermined amounts of pulp from the bulk leach test (CN-4RL) were contacted with predetermined amounts of carbon to achieve various target solution/carbon ratios (or target carbon loadings). The slurries were agitated in rolling bottles with carbon for seven days to "equilibrate" the activated carbon with the gold in solution, after which time the carbon was recovered by screening. Samples of the slurry are filtered to generate carbon and solution assays. The test results are in presented in Figure 13-13. It is important to stress that the isotherm shown in the figures does not represent true equilibrium between gold in solution and on the carbon, but is rather a pseudo equilibrium isotherm that is representative of a practical loading limit in an operating plant. In practice, gold cyanide loading on activated carbon is so slow when it is close to equilibrium that the attainment of true equilibrium takes weeks or even months. These higher loading values are of little relevance as far as plant operations are concerned.

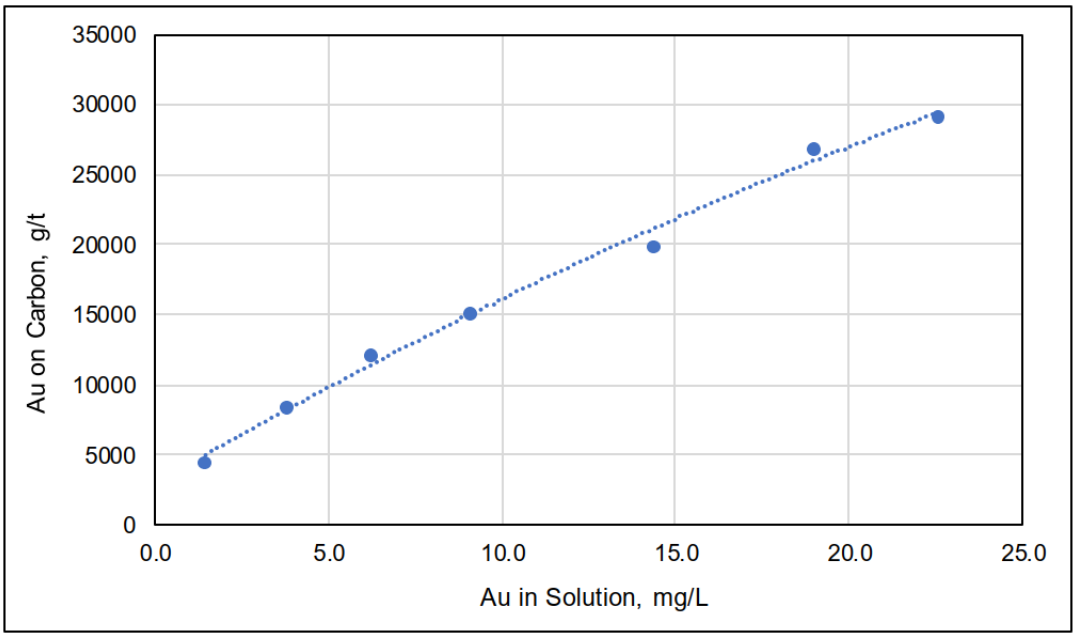


Figure 13-13: Equilibrium Loading of Gold on Carbon

The equilibrium results yielded carbon loading values of up to approximately 30,000 g/t Au, that is beyond the limits that will likely be targeted in the actual CIP plant. The flotation concentrate sample used for this testwork contained approximately 55 to 60 mg/L of gold in the pregnant leach solution, which is high. Trade-off studies would be required to determine a practical (plant) carbon loading, carbon transfer rate, and elution circuit size.

13.2.3.7 Leaching Conclusions and Recommendations

The metallurgical test results were positive and confirmed that high grade gravity concentrate can be directly leached and high gold extractions (about 99%) are achievable. Further testwork is recommended to optimize the cyanide and LeachAID addition.

The flotation concentrate leach results indicated that a gold extraction of approximately 92% can be achieved when regrinding to a grind size P_{80} of 27 µm and using a 72-h leach retention time. However, the tank sizes required for this would be large. Further testwork is recommended to optimize the flotation concentrate leach conditions to maximize gold extraction. The tests could include further optimization of the grind size, cyanide addition, and potentially the addition of lead nitrate. The leach test results indicated that the CIP circuit will have a high feed gold concentration of approximately 55 to 60 mg/L Au in solution. Trade-off studies are recommended to optimize the target carbon loading and elution circuit size.

These results represent a worsening scenario when compared to SGS (Crary, 2017) testwork in Section 13.1.8.4. This further backed up the conclusion reached during this 1,200 t/d expansion study that on-site leaching was not cost effective.

13.2.4 Tailings Testwork Results (2020)

Outotec (Canada) Ltd. conducted thickening testwork on two gold flotation samples, final flotation tailings and rougher concentrate produced by the current Sugar Zone operations.

The purpose of this testwork was to examine the thickening properties of these samples and to determine the operating conditions of the thickeners for expansion study purposes (Haavisto, 2020). The test results are used to recommend and size thickeners, and to ascertain power requirements for future equipment bid purposes.

13.2.4.1 Objective of Tests and Selected Test Equipment

The purpose of the testing was to conduct thickener testwork on a two gold flotation tailings samples, with the objectives to determine:

- Coagulant and flocculant type and dose
- Overflow clarity
- Underflow density
- Underflow yield stress.

A 99 mm diameter Supaflo high-rate thickener test rig was used in this test campaign.

13.2.4.2 Sample Characterization

The solids from the slurry samples and the underflow samples that were generated from each thickening test were dried overnight in an oven at 40°C. The suspended solids in the overflow samples collected from each test were determined by filtering 100 mL of overflow liquor through pre-weighed 47 mm Millipore Express Plus 0.22 PES Membrane filter discs using a vacuum filtration unit. The collected overflow solids were also washed thoroughly prior to oven drying at approximately 40°C. The filter discs were then reweighed to determine the suspended solids content.

Representative samples of the feed slurries were submitted for laser and dry screen sizing.

13.2.4.3 Flocculant and Coagulant Selection

Harte Gold Corp. operations had requested that current on-site reagents and alternative reagents from preferred suppliers be used for the majority of tests. Based on this request, and static cylinder tests, the flocculants and coagulants selected for the final tailings dynamic testwork were POLYCLEAR 2501 paired with FLOQUAT FL3050 and ZETAG 4105 paired with AMERFLOC 485. For the rougher concentrate dynamic testwork, the flocculants selected were POLYCLEAR 2501 and ZETAG 4105. A range of Solenis, Quadra, and SNF products were used for the flocculant selection testwork. This should not be viewed as an endorsement of these particular suppliers, and other equivalent products may be used.

The flocculants used for the screening were 905 VHM, 913 SH, 920 VHM, and 934 VHM from SNF, POLYCLEAR 2501 and ZETAG 4105. The coagulants used for the screening were FLOQUAT FL3050 and AMERFLOC 485. Further flocculant screening may achieve better results.

13.2.4.4 Feed Dilution Tests

Dilution tests were conducted using Outotec's optimum dilution test method. Outotec's optimum dilution method finds the optimum feed solids concentration for use with Outotec's Vane Feedwell. Full dilution data for these tests can be found in thickening report (Haavisto, 2020).

13.2.4.5 Batch Dynamic Thickening Tests

The samples were dynamically thickened in the 99 mm lab rig. To determine the thickener rake drive size, testing was conducted to determine the yield stress.

For each test, stock slurry was drawn from an agitated vessel by a variable-speed peristaltic pump and diluted as required "on the fly" using a second pump.

Flocculant was added at a set flow rate, but at a range of concentrations to designed deliver the required dosage for each test. Stock flocculant solution was hydrolyzed in tap water to 0.25% w/v (2.5 g/L), then diluted as required. The diluted flocculant was dosed into the diluted line downstream of coagulant dosing.

Coagulant was added at a set flow rate, but at a range of concentrations designed to deliver the required dosage for each test. Stock coagulant solution was hydrolyzed in tap water to 0.10% w/v (1.0 g/L), then diluted as required. The diluted coagulant was dosed into the diluent line.

The flocculated solids bed was raked continuously at 2 rpm to de-water. For the tailings sample, two rotating pickets were used in conjunction with two stationary pickets. For the concentrate sample, rotating pickets were not used.

Overflow samples were collected at a bed height of 160 mm just at the opening of the feedwell discharge. The samples were analyzed to determine the suspended solids content.

Underflow samples were taken using a positive displacement pump. For the tailings sample, this sample was taken at a bed height of 240 mm. For the concentrate sample, this was taken at bed height of 160 mm. The solids concentration was then determined.

All rheological measurements were carried out using a Thermo Haake VT550 rheometer and an OK600 4-blade vane. The method measures shear stress versus time, with the peak of this curve equating to the yield stress. A constant shear rate of 0.1 sec-1 was used. For each dynamic test underflow sample, a simple unsheared vane yield stress was measured.

The underflow yield stress is measured to generate data for use in the sizing of the thickener rake drive, it is not intended to be used for any other purpose.

13.2.4.6 Test—Product Data

Feed final flotation tailings sample was provided as slurry in eight 20 L pails. Feed rougher concentrate sample was provided as slurry in eight 20 L pails. Process water was decanted from all the pails. The information related to each product is shown in Table 13-39 and Table 13-40 for Final flotation tailings and rougher concentrate, respectively.

Table 13-39: Final Flotation Tailings Data

Type of Process/Product	Final Flotation Tailings (FT)	
Slurry as Tested—Pre-leach	Temperature	Ambient (20°C)
	Density	1.49 kg/L
	Solids SG	2.91 t/m^3
	pH	7.7
	Sizing P_{80}	80 µm
	Solids phase Composition	Silicates
	Liquid phase Composition	Water
	Corrosive Elements	N/A

Table 13-40: Rougher Concentrate Data

Type of Process/Product	Rougher Concentrate (RC)	
Slurry as Tested—Pre-leach	Temperature	Ambient (20°C)
	Density	1.40 kg/L
	Solids SG	3.96 t/m^3
	pH	8.5
	Sizing P_{80}	63 µm
	Solids phase Composition	Sulphides
	Liquid phase Composition	Water
	Corrosive Elements	N/A

13.2.4.7 Flocculant Screening—Final Flotation Tailings

The flocculant that had the best overall performance was ZETAG 4105. It had the fastest settle rate, with clear overflow at the lower dosages, although it over-flocculated at a dose of 20 g/t.

SNF 934 VHM did not produce a clear overflow.

POLYCLEAR 2501, SNF 913 SH, and SNF 920 VHM did not produce a clear overflow and settling was not visible.

SNF 913 SH, 920 VHM, and 934 VHM were not tested further.

The flocculant screening settle rate curves for the different flocculants tested for flotation tailings are presented in Figure 13-14.

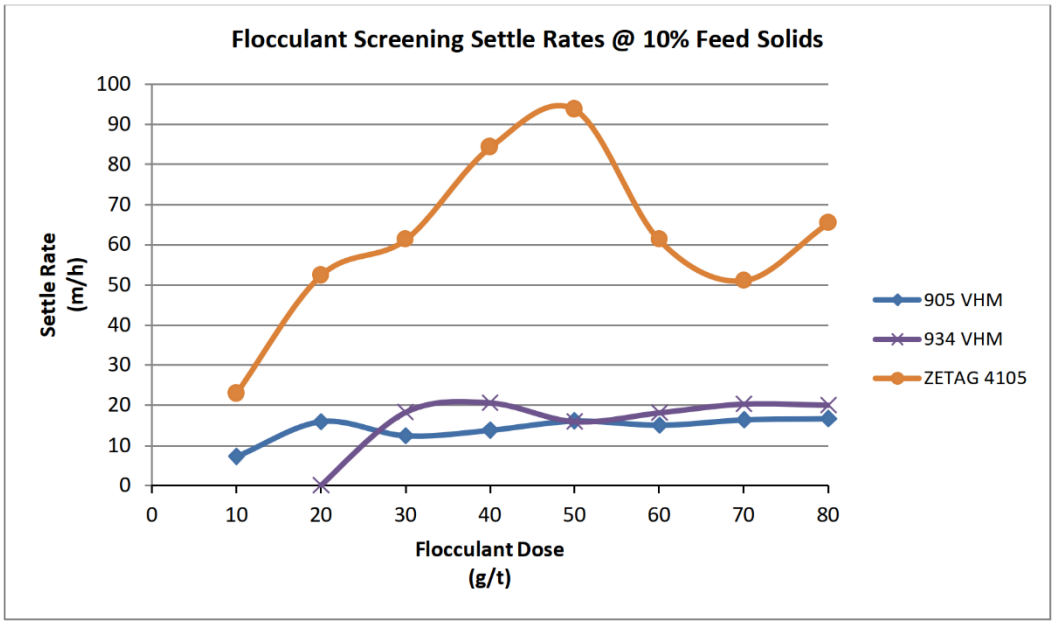


Figure 13-14: Flotation Tailings—Flocculant Screening Results

The flocculant and coagulant that had the best overall performance were ZETAG 4105 and AMERFLOC 485, respectively. They have the fastest settle rate with clear overflow at the lower dosages, although they over-flocculate. Both combinations of reagents were used in dynamic testwork.

The flocculant and coagulant screening settle rate curves for the different reagents tested for flotation tailings are presented in Figure 13-15.

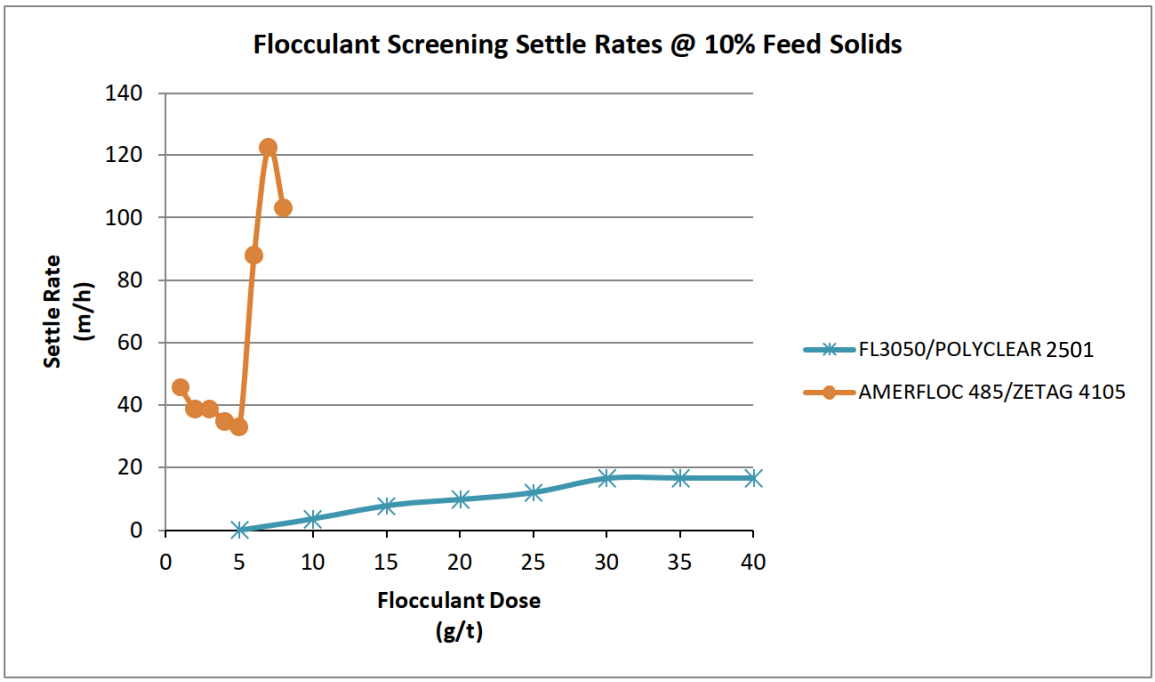


Figure 13-15: Flocculant Screening Results at 37 g/t Coagulant

13.2.4.8 Flocculant Screening—Rougher Concentrate

The flocculant that had the best overall performance was ZETAG 4105. It has the fastest settle rate with clear overflow at the lower dosages and was used in all dynamic testwork along with POLYCLEAR 2501.

The flocculant screening settle rate curves for the different flocculant tested for rougher concentrate are presented in Figure 13-16.

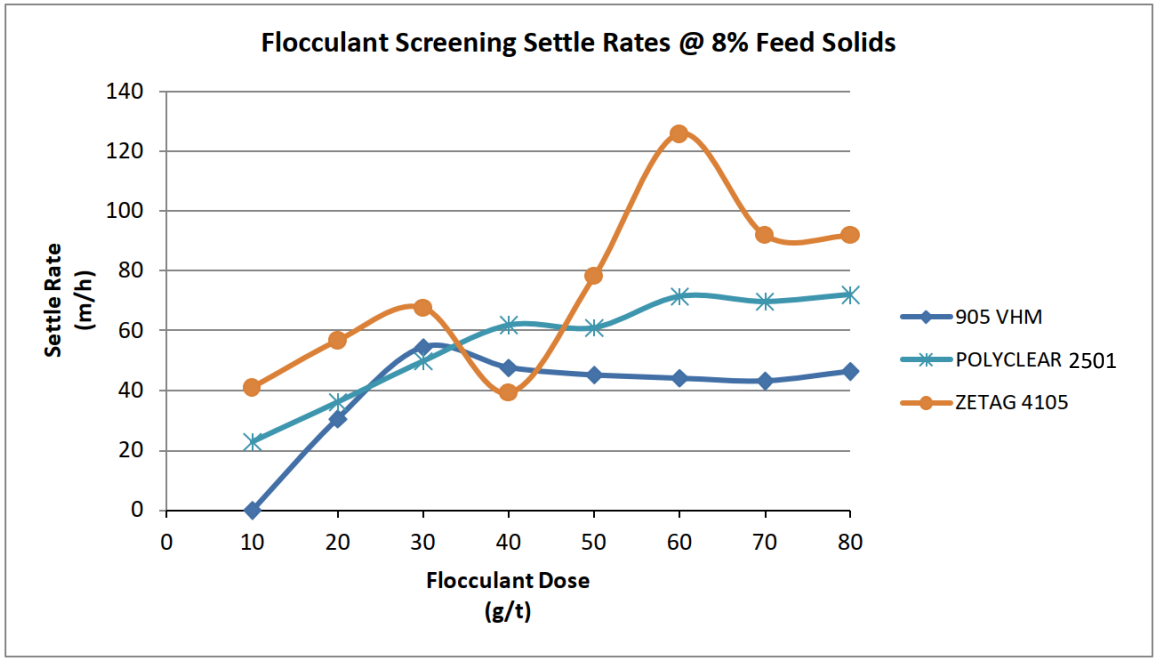


Figure 13-16: Rougher Concentrate—Flocculant Screening Results

13.2.4.9 Dynamic Thickening Tests—Flotation Tailings

Multiple runs were conducted on the gold flotation tailings samples. These runs were conducted at different flocculant dosages and solids loading rate to determine impact on the underflow densities and overflow clarity and to examine the performance of the existing thickener as well as potential retrofits.

The test results for the final flotation tailings sample are summarized in Table 13-41. Test results for the current reagents—FLOQUAT FL3050 and POLYCLEAR 2501—are depicted within the thickening report Haavisto (2020). Figure 13-17 and Figure 13-18 illustrate the effects of flocculant dosage and solids loading rate on underflow density and yield stress, respectively. Photographs of the overflow clarities are shown within the thickening report Haavisto (2020).

Table 13-41: Final Flotation Tailings Dynamic Thickening Test Results

Feed		Coagulant		Flocculant		Underflow		Overflow Solids (mg/L)
Flux ($t/(m^2 \cdot h)$)	Liquor RR (m/h)	Type	Dose (g/t)	Type	Dose (g/t)	Meas. Solids (% (w/w))	YS (Pa)	
0.35	2.27	FLOQUAT FL3050	37	POLYCLEAR 2501	4	72.0	28	126
0.35	2.27				10	73.3	77	100
0.45	2.92				10	69.8	41	223
0.35	2.27	AMERFLOC 485	37	ZETAG 4105	4	72.4	38	223
0.35	2.27				10	72.9	58	129
0.35	2.27				20	66.3	16	127
0.45	2.92				10	72.6	50	225

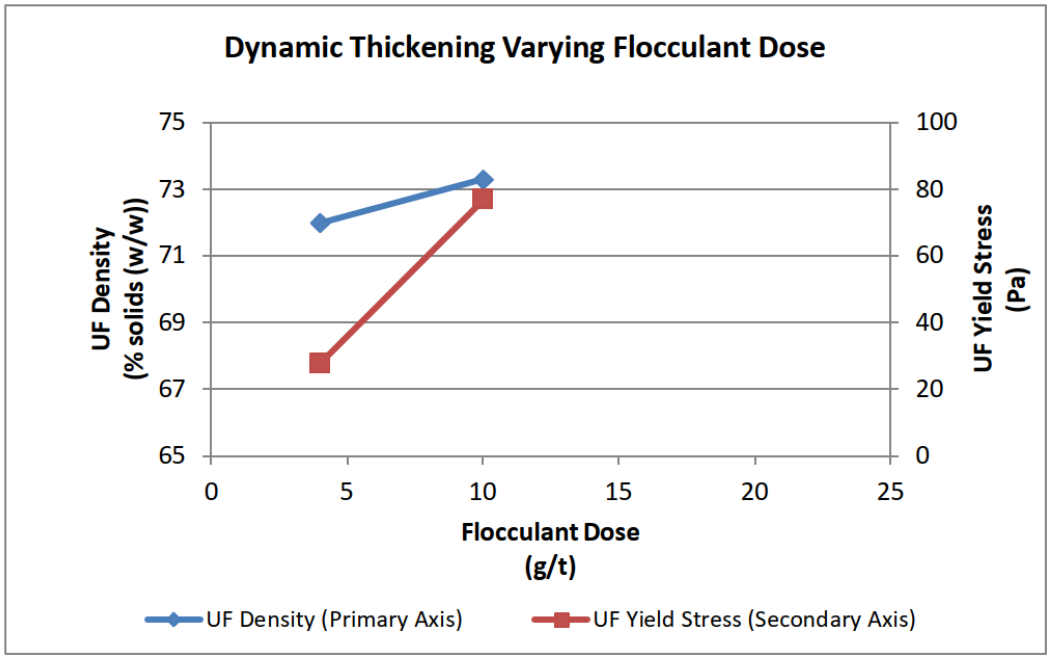


Figure 13-17: Effect of POLYCLEAR 2501 Flocculant Dosage on Underflow Density for Flotation Tailings at 0.35 t/(m²·h) with 37 g/t FLOQUAT FL3050 Coagulant Dosage—Current REAGENTS

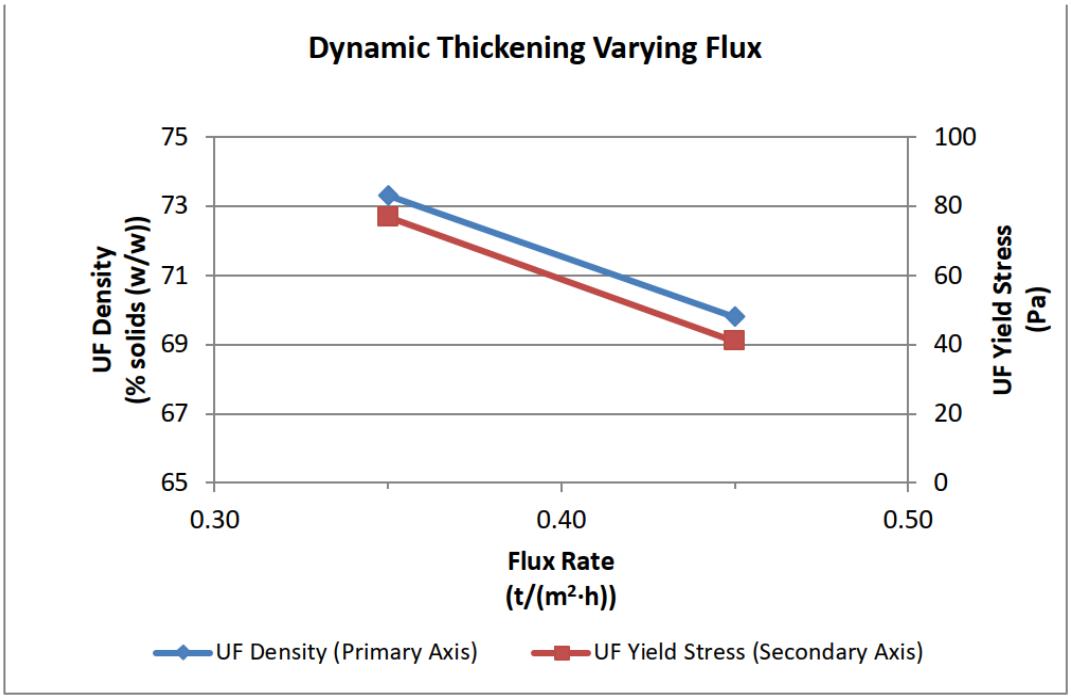


Figure 13-18: Effect of Solids Loading Rate on Underflow Density for Flotation Tailings using POLYCLEAR 2501 and FLOQUAT FL3050

Test results for the alternative reagents—ZETAG 4105 and AMERFLOC 485—are depicted within the thickening report Haavisto (2020). Figure 13-19 and Figure 13-19 illustrate the effects of flocculant dosage and solids loading rate on underflow density and yield stress, respectively. Photographs of the overflow clarities are shown within the thickening report (Haavisto, 2020).

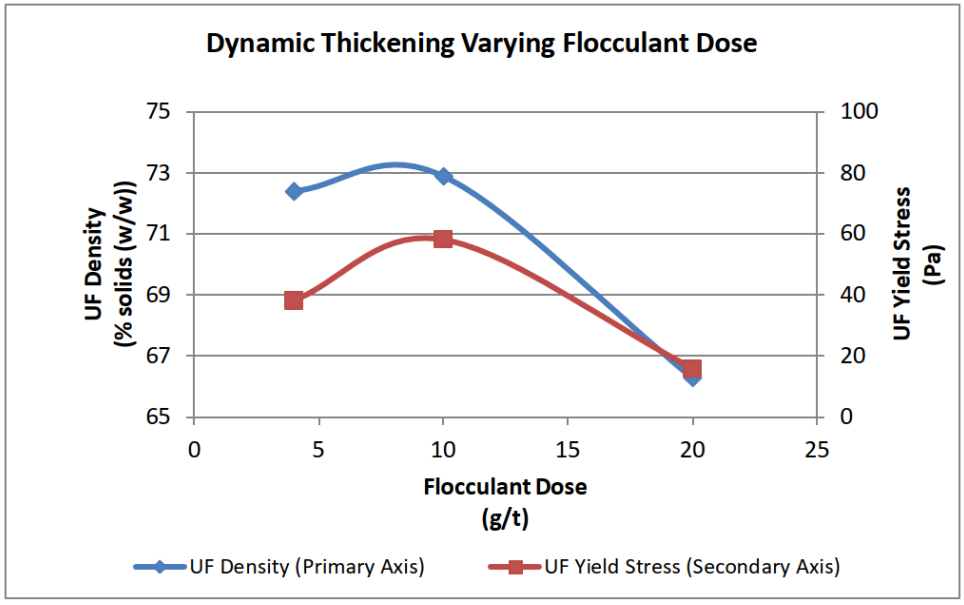


Figure 13-19: Effect of ZETAG 4105 Flocculant Dosage on Underflow Density for Flotation Tailings at 0.35 t/(m²·h) with 37 g/t AMERFLOC 485 Coagulant Dosage

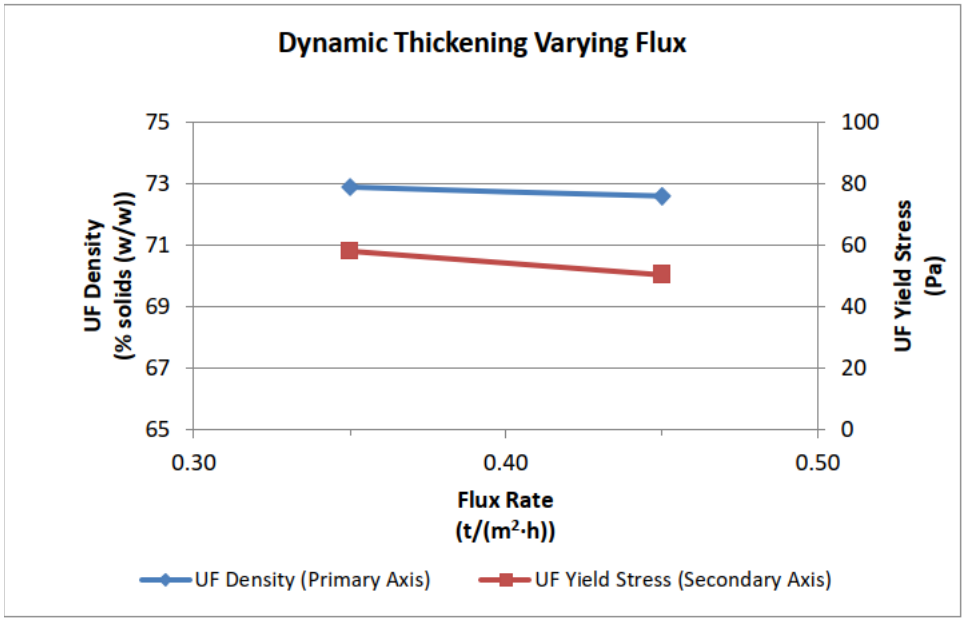


Figure 13-20: Effect of Solids Loading Rate on Underflow Density for Flotation Tailings using ZETAG 4105 and AMERFLOC 485

Overflow clarity for all dynamic test runs was below 250 ppm, and suitable for process water reuse. All runs surpassed the desired target underflow density requested by Harte Gold operations of 65% solids w/w.

Reagent dose rate optimization was performed with varying flocculant dosages to examine the change in underflow and overflow at the current feed rate to the existing 12 m thickener resulting in a solid loading rate of 0.35 t/(m^2·h). Test Runs were performed at increased solids loading rates at optimized reagent dosage rates to test the target 50 t/h to the existing 12 m thickener resulting in a solid rate of 0.45 t/(m^2·h). The highest density achieved at the target feed rate to the existing thickener was 72.6% solids (w/w) at a solid loading rate of 0.45 t/(m^2·h) using ZETAG 4105 and AMERFLOC 485.

13.2.4.10 Dynamic Thickening Tests—Rougher Concentrate

Multiple runs were conducted on the gold flotation samples. These runs were conducted at different flocculant dosages and solids loading rates to determine impact on the underflow densities and overflow clarity and to examine the performance of the existing thickener, as well as potential retrofits.

The test results for the rougher concentrate sample are summarized in Table 13-42. Test results for the current flocculant—POLYCLEAR 2501—are depicted in Figure 13-21 and Figure 13-22, for the effects of flocculant dosage and solids loading rate on underflow density and yield stress, respectively. Photographs of the overflow clarities are shown in the thickening report Haavisto (2020).

Table 13-42 Rougher Concentrate—Dynamic Thickening Test Results

Run No.	Feed		Flocculant		Underflow		Overflow Solids (mg/L)
	Flux (t/(m^2·h))	Liquor RR (m/h)	Type	Dose (g/t)	Meas. Solids (% (w/w))	YS (Pa)	
1	0.16	1.03	POLYCLEAR 2501	20	75.0	47	2,883
2[1]	0.16	1.03		40	71.7	93	263
3[1]	0.16	1.03		30	72.7	77	264
4[1]	0.08	0.52		30	73.1	58	<100
5	0.16	1.03	ZETAG 4105	30	76.4	111	
6	0.16	1.03		20	76.4	87	
7	0.08	0.53		20	79.9	221	
8	0.25	1.61		20	75.2	51	

Notes: [1] Reduced bed height.

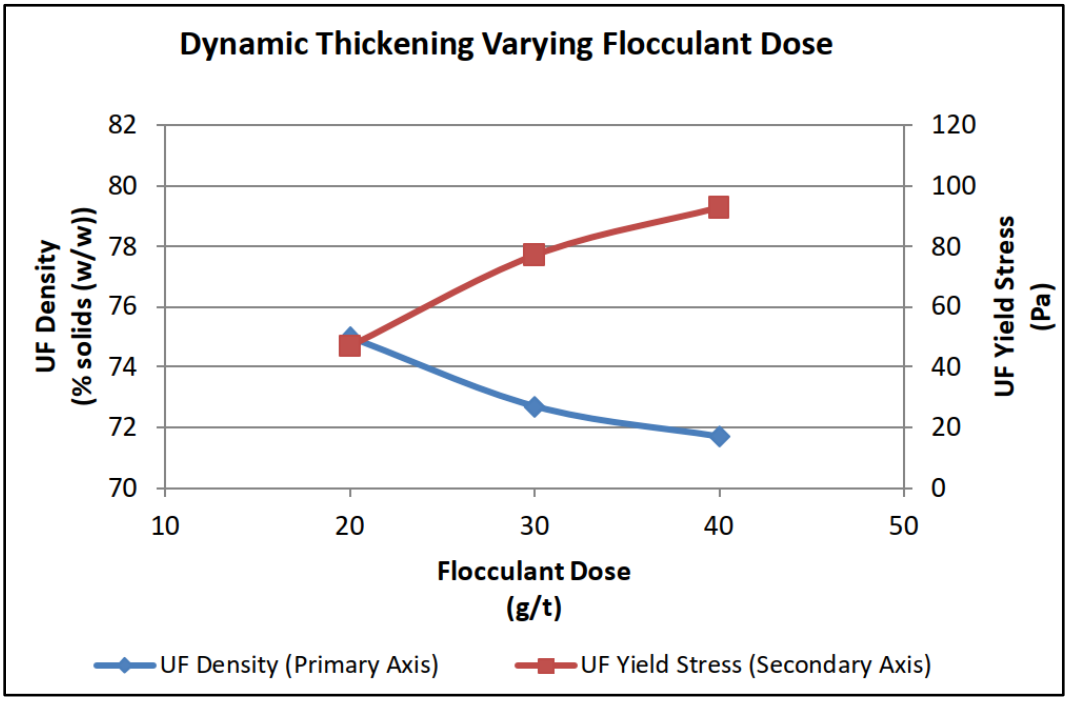


Figure 13-21: Effect of Flocculant Dosage on Underflow Density for Rougher Concentrate at 0.16 t/(m²·h) using POLYCLEAR 2501

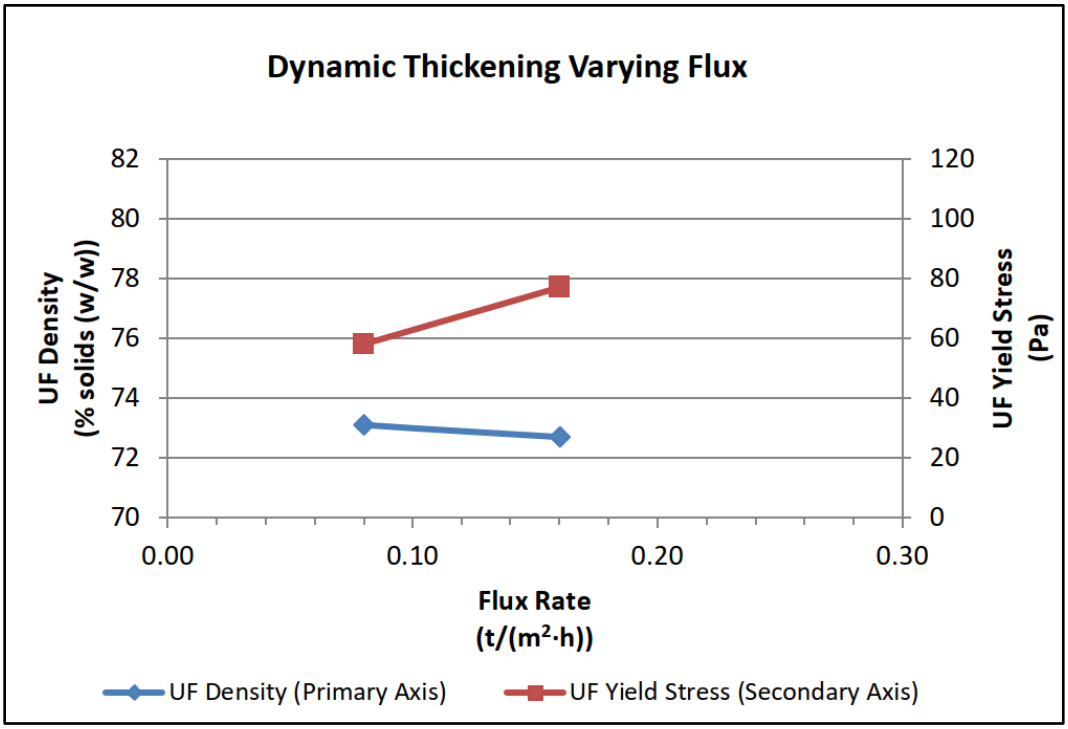


Figure 13-22: Effect of Solids Loading Rate on Underflow Density for Rougher Concentrate using POLYCLEAR 2501

Test results for the alternative flocculant—ZETAG 4105—are depicted in Figure 13-23 and Figure 13-24, for the effects of flocculant dosage and solids loading rate on underflow density and yield stress, respectively. Photographs of the overflow clarities are shown in the thickening report Haavisto (2020).

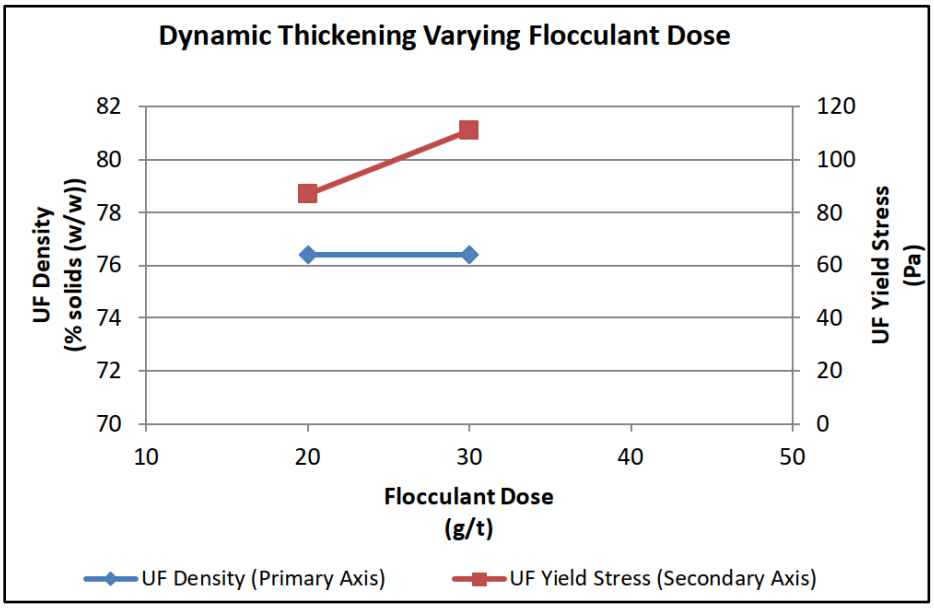


Figure 13-23: Effect of Flocculant Dosage on Underflow Density for Rougher Concentrate at 0.16 t/(m².h) using ZETAG 4105

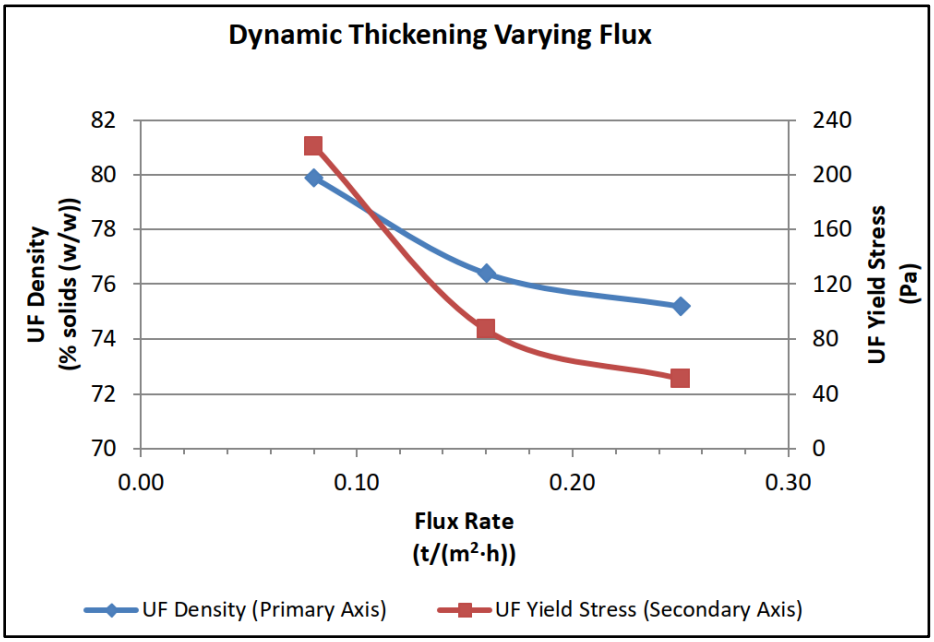


Figure 13-24: Effect of Solids Loading Rate on Underflow Density for Rougher Concentrate using ZETAG 4105

All dynamic test runs using POLYCLEAR 2501 generated significant froth, had cloudy overflow clarity at flocculant dose rates at or below 20 g/t, and reduced bed heights at flocculant dose rates of 30 g/t or above.

All dynamic test runs using ZETAG 4105 generated froth. Overflow clarity for all dynamic test runs using ZETAG 4105 was below 250 ppm and suitable for process water reuse.

All runs surpassed the desired target underflow density according to the questionnaire of 65% solids w/w.

Flocculant dose rate optimization was performed at varying flocculant dosages to examine the change in underflow and overflow at a solid loading rate of 0.16 t/(m^2•h). POLYCLEAR 2501 had peak underflow density and the clearest overflow at 30 g/t, while ZETAG 4150 had peak underflow and clear overflow at 20 g/t.

Test runs at varying solids loading rates were performed using these optimized flocculant dose rates. The highest density achieved at the target feed rate of 1 t/h to the current 4 m thickener was 79.9% solids w/w at a solid loading rate of 0.08 t/(m^2·h) using ZETAG 4105.

An increased solid loading rate of 0.25 t/(m^2·h) was tested to determine the changes to underflow and overflow at higher feed rates. The underflow density at this increased rate was 75.2% solids w/w using ZETAG 4105.

13.2.4.11 Conclusion

The testwork on the Harte Gold Corporation Sugar Zone final flotation tailings sample has shown that the material can achieve 72.6 (% solids (w/w)) while using 10 g/t ZETAG 4105 flocculant and 37 g/t AMERFLOC 487 coagulant at a solids loading rate of 0.45 t/(m^2·h).

The testwork on the Harte Gold Corporation Sugar Zone rougher concentrate sample has shown that the material can achieve 80.1% solids (w/w) while using 20 g/t ZETAG 4105 flocculant at a solids loading rate of 0.08 t/(m^2·h).

All of the results are based on the samples as tested, with conditions specified in the Tailings Thickening report (Haavisto, 2020).

13.3 Basis and Trade-Offs for Comminution and Recovery Circuit Design

13.3.1 Introduction

The three flowsheets considered in this 1,200 t/d expansion study were:

1. Gravity concentration followed by rougher flotation with cleaner flotation and sale of concentrate

2. Gravity Concentration followed by rougher flotation and leaching with Merrill-Crowe (MC) gold recovery of the reground concentrate to keep gold production on site
3. Gravity concentration followed by rougher flotation and leaching with CIL gold recovery of the reground concentrate to keep gold production on site.

To minimize capital costs and simplify design and operations, the Harte Gold process plant was initially designed to produce a gravity concentrate, for on-site treatment, and a rougher flotation concentrate for sale.

The gold recoveries obtained during the 2020 test program suggest a significant difference in leaching kinetics compared to previous SGS programs (Crary, 2017, 2019).

13.3.2 Summary

Following a simple trade-off, the original flowsheet was retained, that is, gravity concentration followed by rougher flotation and sale of a concentrate over 100 g/t.

SGS has carried out six sample test programs between 2010 and 2021. Cyanide leaching of flotation concentrates was tested in five of the programs: 2010, 2015, 2017, 2019, and 2021.

The 2019 testwork report showed that concentrate leaches well, particularly when reground to 25 to 35 µm and leached for 48 h, with a relatively high cyanide addition. After allowing for potential soluble gold losses, a recovery of 97% has been estimated. However, the 2020 flotation concentrate leach results indicate that a gold extraction of only approximately 92% can be achieved when regrinding to a grind size P_{80} of 27 µm and using a 72-h leach retention time. These results show a significant decrease in leaching gold recovery, and increase in required leaching retention time, which could be attributed to ore variability; therefore, it is recommended that additional testwork should occur before leaching is pursued in the future.

Based on this return-on-investment figures, the preferred option is to continue to operate the plant in the current manner, as constructed, and to add a cleaner flotation cell to the current flowsheet.

13.3.3 SGS Cyanidation Test Results

This section contains a summary of the metallurgical results from the SGS report (Crary, 2021, January 14.

13.3.3.1 Test Results (2010)

A bulk flotation test was conducted on the gravity tailing to generate a rougher concentrate for gold leaching. Cyanidation of the reground rougher flotation concentrate at a P_{80} of 21 µm achieved gold and silver recoveries of 98.6% and 83.8%, respectively.

13.3.3.2 Test Results (2015)

Two flotation concentrate cyanidation tests were completed using the concentrate: the "as-is" sample and a reground sample. The concentrate produced from the master composite responded well to cyanidation, and after 48 h of leaching the gold leach extractions were between 97.4% and 99.1%. Regrinding the concentrate increased the overall gold extraction, and slightly increased the cyanide and lime consumption.

13.3.3.3 Test Results (2017)

The tests were conducted in standard bottle rolls, and subsamples of the leach solution were taken at specific time intervals and assayed for gold to monitor the rate of extraction. The flotation concentrates responded well to cyanidation, and after 48 h of leaching, the gold extractions were approximately 96% to 99%. Regrinding the concentrate produced from the monthly master composite increased overall gold extraction by 1.5%, and also increased cyanide consumption.

13.3.3.4 Test Results (2019)

Four flotation concentrate cyanidation tests were completed using as-received plant samples, and two variability sample tests were completed using the combined concentrate generated from tests F-6 to F-9. The objective of the tests was to determine the impact of regrinding, and to establish the gold extraction (and overall gold recovery) for gravity + flotation + cyanidation flowsheet.

The tests were conducted in standard bottle rolls, and subsamples of the leach solution were taken at specific time intervals and assayed for gold to monitor the rate of extraction. Oxygen was sparged into the bottles to maintain the DO concentration target. To assess the impact on solution chemistry, the variability samples include a pre-aeration stage before cyanidation.

The flotation concentrates responded well to cyanidation, and after 48 h of leaching the gold extractions were approximately 96% to 99%. Regrinding the concentrate produced from the monthly master composite increased the overall gold extraction by 1.5% to almost 3%, but also increased the cyanide consumption.

13.3.3.5 Test Results (2021)

One gravity concentrate and one flotation concentrate sample were received at SGS Lakefield for a metallurgical test program. Gravity concentrate was submitted for two intensive cyanidation tests, and flotation concentrate was submitted for a series of tests that provided data for preliminary design of the leach and carbon adsorption circuits. The gravity concentrate sample contained approximately 5,113 g/t Au (calculated head from test program). The flotation concentrate contained 112 g/t Au, 63 g/t Ag, and 25.3% $S^{=}$ (direct head assays).

The gravity concentrate leach results indicated that high gold extractions (~99%) can be achieved when using intensive cyanidation conditions (high cyanide, DO, and LeachAID addition).

The flotation concentrate leach results indicated that a gold extraction of approximately 92% can be achieved when regrinding to a grind size P_{80} of 27 µm and using a 72-h leach retention time. The

sample continued to leach throughout the test, which used 1 g/L NaCN and had a DO concentration of approximately 25 mg/L.

The leach test results indicated that the CIL circuit will have a feed solution of approximately 55 to 60 mg/L Au. Carbon contact tests were completed, and the results will be used by Harte Gold on an initial CIL plant design.

13.3.4 Justification for Second Ball Mill and Upgrade of Cone Crusher

OMC (Festa, 2020, October) undertook modelling with simulations on the comminution circuit in a two-phase approach. In the Phase 1 report—comminution performance review (Festa, 2020, December)—the operating data analysis was undertaken to provide four options to consider for expanding the milling circuit. In the Phase 2 report—comminution expansion to 1,200 t/d)—the modelling of the selected option and recommendations were provided.

The BWi index is a critical element for determining mill size. In general, the closing screen size used for a BWi test affects the result such that the smaller the screen size, the higher the BWi result. It is therefore recommended that to be able to compare BWi tests on various ores, the tests be carried out at the same closing screen size for a fair comparison. Given the target grind size P_{80} of 75 µm, a 106 µm closing screen size should be used for all future BWi tests.

The initial design point BWi of 16.1 kWh/t was obtained from a 2013 composite from surface trenches. A BWi of 12.1 kWh/t figure was obtained from a 2010 composite from within the bulk sample zone. Seven 2017 monthly composites from the bulk sample zone averaged 14.7 kWh/t (75 µm screen). Recent ball mill feed composites tested at 9.9 kWh/t (106 µm screen) to 11.6 kWh/t (150 µm screen), much softer than the design point 2013 composite of 16.1 kWh/t.

If the records allow for it, the geology and rock types of the bulk sample Zone 2017 composites should be compared to those of the zone or zones where the recent ball mill feed composites were mined, to try to ascertain why the results were lower. The answer may lie in the fact that site operations suggest that the mine has greatly reduced the dilution from the HW and FW metavolcanics, thus reducing the hardness of the ore.

It is also noted that no testwork of the deeper ore has been carried out, and it is recommended that a variability sampling and testwork program be carried out, which should align to the LOM schedule. This is true of the Wolf Zone and other recently discovered zones. The samples should allow for SMC, RWi, BWi, and Ai testwork. Ideally, all zones of LOM schedule should be included if the design point BWi value is to be further tested.

While the opportunity exists for further scrutiny of the BWi index, the decision was made to continue to use the design point BWi of 16.1 kWh/t for the study. A summary of the BWi testwork results is shown in Table 13-43 for testwork from 2010 to 2020.

The crushing work index of 12.7 kWh/t was used for crusher simulation, along with a check of the operational data power usage at the maximum value of 21.8 kWh/t.

Table 13-43: BWi, Ai, and RWi of Sugar Zone Composites Summary

Description	Unit	2010 Comp Upper and Lower Zone	2013 Comp Lower Trench (Design)	2017 May/Jun Comp	2017 Jul Comp	2017 Aug Comp	2017 Sep Comp	2017 Oct Comp	2017 Nov Comp	2017 Dec Comp	2019 Nov BMF Comp		2020 Mar Comp	2020 Aug Comp		
Screen Mesh		-	140	200	200	200	200	200	200	200	100	140	100	100	140	100
Screen Aperture	µm	-	106	75	75	75	75	75	75	75	150	106	150	150	106	150
Feed F_{80}	µm	-	2,480	2,430	2,360	2,346	2,284	2,264	2,238	2,277	2,275	2,275	2,275	2,123	2,123	1,943
Product P_{80}	µm	-	91	63	62	61	62	60	62	60	107	77	107	104	79	110
gpr	g/rev	-	1.29	1.22	1.18	1.20	1.26	1.15	1.19	1.15	2.51	2.09	2.39	2.54	1.96	2.11
BWi	kWh/t	12.1	**16.1**	14.6	14.9	14.6	14.2	15.0	14.9	15.0	9.61	9.86	10.01	9.46	10.65	11.57[1]
RWi	kWh/t	-	**11.5**	-	-	-	-	-	-	-	-	-	-	-	-	-
Ai	g	-	**0.616**	0.563	0.587	0.601	0.665	0.677	0.652	0.631	-	-	-	-	-	-
SMC A x b		-	**57.5**	-	-	-	-	-	-	-	-	-	-	-	-	-

Notes: [1] ALS QC.

The Ai average is 0.624 g, and is consistent with the design point of 0.616 g from the 2013 composite used as part of the original plant design parameters, shown in Table 13-44.

Table 13-44: Design Ore Characteristics from 2013 Composite

Parameter	Unit	Design	Source
Crushing Work Index (CWi)	kWh/t	12.7	SGS
Bond Work Index 106 µm Screen (BWi)	kWh/t	16.1	SGS
Rod Work Index (RWi)	kWh/t	11.5	SGS
Axb	-	57.5	SGS
Abrasion Index (Ai)	g	0.616	SGS
Specific Gravity	-	2.88	Owner

Mill simulations showed that additional milling power was required to process the increase in throughput from 800 to 1,200 t/d. A 550-kW ball mill was chosen and used in simulations.

Operational data of large crushing (more than 1,000 t/d throughput) showed that the HP100 cone crusher tripped on power demand on two of three occasions. Additionally, other operational issues have led to the decision to install a new HP200 Cone Crusher, as the costs and downtime needed to upgrade the existing HP100 structure (including the feed bin and feeder above the HP100) were considered too costly. The HP100 would remain in place and assist in operations when the HP200 circuit was down for maintenance, thus adding operational flexibility to the crushing area, albeit at a reduced crushing throughput.

13.3.5 Justification for Increasing the Current Circuit Capacity

A preliminary economic analysis comparing concentrate sale with concentrate leaching was carried out using the actual concentrate sale terms, the SGS leach data, leach plant capital, OPEX estimates from Halyard, and recent plant operating data. It is summarized below.

13.3.5.1 Concentrate Sale

For the concentrate sale option, it is expected that the process plant will continue to be operated at a lower flotation mass pull, that is, with a reduced mass of higher-grade concentrate. Depending on the ore grade and the recovery split between the gravity and flotation concentrates, the concentrate is estimated to increase from approximately 100 g/t Au produced at an approximate 2.5% mass pull, to approximately 130 g/t Au produced at a similar overall flotation mass pull.

It is typical that the flotation concentrate grade is approximately 100 g/t.

The current contractual terms are presented in Table 13-45.

Table 13-45: Concentrate Contractual Terms

Transport Costs	C$/wt	113.75
	US$/wt	84.26
Typical Moisture Penalty	US$/wt	5.00
Gold Price	US$/oz	1,700
Payables <50 g/t Au	%	93
50 g/t–75 g/t Au	%	95
75 g/t–100 g/t Au	%	96
>100 g/t Au	%	96.5
Treatment Cost	US$	180.00
Refining Cost	US$/oz Au	8.00

13.3.6 Economic Evaluation

Using data obtained in the flowsheets of the trade-off study (Halyard, 2021), an analysis was prepared comparing the economics of the three options: gravity concentration followed by rougher flotation with cleaner flotation and sale of concentrate; gravity concentration followed by rougher flotation and leaching with Merrill-Crowe (MC) gold recovery of the reground concentrate to keep gold production on site; and gravity concentration followed by rougher flotation and leaching with CIL gold recovery of the reground concentrate to keep gold production on site.

It was determined that the CIL option, at a capital cost estimate of $29.25 million, was the least cost-effective option, followed by the MC option at a capital cost estimate of $25.15 million. Considering that both had an anticipated recovery of approximately 94.2% the cost-to-benefit ratio did not compare well to the expansion of the current gravity gold recovery and expanded flotation circuit with added cleaner flotation and concentrate sale, at an anticipated capital cost estimate of $16.35 million and a recovery of approximately 94.4%.

13.4 Recommended Flowsheet, Recoveries, and Design Criteria

Various flowsheets have been considered and tested during Project development, based on data from the test programs and other sources.

For the expansion to 1,200 t/d throughput, the selected flowsheet was gravity concentration producing a doré gold bar, and rougher flotation plus cleaner flotation to produce gold concentrate. This was because recoveries and revenues obtained by gravity and flotation only compared well with whole-ore cyanidation. There are also obvious advantages in avoiding the permitting and use of cyanide on site.

In the expansion study, the recovery of gold to a gravity concentrate was estimated to be approximately 62.5%. Of this amount, 82% was estimated to be recovered as doré, by tabling and smelting the gravity concentrate. Not all the gravity concentrate is recovered as doré; the table tailings

will be added to the flotation concentrate and the middlings will likely be recirculated. The overall recovery by gravity, and rougher and cleaner flotation, is estimated at 94.4%. Of this, 62.5% is estimated to be as doré, from smelting of the gravity concentrate, and 31.9% expected to report to the flotation concentrate.

The plant was designed, constructed, and commissioned in late 2018 based on this flowsheet. Table 13-46 shows the actual recoveries and grades obtained from plant operation in 2019 and 2020:

Table 13-46: Actual Plant Results

	2019	2020
Mill Feed Grade (g/t)	4.5	5.7
Gravity Recovery (%)	65.6	70.4
Flotation Recovery (%)	27.1	23.2
Overall Recovery (%)	92.7	93.6

Recent operation data show some variable results for the concentrate grade, as follows:

- October 2020—mill feed, 9.26 g/t; gravity, 66.4 %; flotation, 28.2%; overall recovery, 94.6%; flotation concentrate grade, 137 g/t
- November 2020—mill feed, 6.68 g/t; gravity, 74.3%; flotation, 21%; overall recovery, 95.3%; flotation concentrate grade, 86 g/t.

- These actual recoveries generally compare well with the Preliminary Economic Assessment (PEA) estimates. The differences in operational data between 2019 and 2020 are because the plant was in a “post-commissioning” period and operations were optimizing the circuits, such as concentrate mass pulls. The mill feed grade was also lower than design, which has been shown to adversely affect recovery by a few percentage points.

Further analyses of the actual operating results show the following trends:

- The overall recovery increases with head grade, but differences are relatively minor for head grades between 5 and 7 g/t. An increase of approximately 1.25% can be expected for each 1 g/t increase in the head grade.
- The relative contribution to overall recovery of gravity and flotation varies with the head grade. The proportion recovered by gravity increases as the head grade increases. At the expected range of head grades, the gravity and flotation components are estimated to be 62.5% and 31.9%, respectively, for a total of 94.4%.

Following a simple trade-off, described above, the selected flowsheet was gravity concentration followed by rougher and cleaner flotation and sale of a concentrate at an estimated 130 g/t.

Table 13-47 shows the estimated recoveries and grades for these flowsheets.

Table 13-47: Estimated Recoveries and Grades

Flowsheet	Gravity/Flotation Medium Grade Conc. Sale (PEA)	Gravity/Flotation High Grade Conc. Sale (FS)[1]	Gravity/Flotation and Conc. Leach by MC	Gravity/Flotation and Conc. Leach by CIL
Mill Feed Grade g/t	6 to 7	6 to 7	6 to 7	6 to 7
Gravity Recovery %	75	62.5	62.5	62.5
Flotation Recovery %	18	31.9	32.7	32.7
Flotation Concentrate Grade g/t	>50	>100	-	-
Conc. Leach Recovery %	-	-	97	97
Overall Recovery %	93	94.4	94.2	94.2

Note: [1] Selected flowsheet.
CIL = carbon in leach; FS = Feasibility Study; MC = Merrill-Crowe; PEA = Preliminary Economic Assessment.

Based on these criteria, it is recommended that apart from optimizing concentrate grade, the process plant could continue to be operated in a similar manner as currently, with only the cleaner flotation cell added.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The following Technical Report section provides a summary of the Mineral Resource estimate for the Harte Gold Sugar Zone Operation. The Mineral Resource estimate presented herein is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 and has been estimated in conformity with the generally accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM, 2019) *CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines*. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into a Mineral Reserve. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Mineral Resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent Mineral Resource estimates.

The effective date of this Mineral Resource estimate is 30 September 2020.

14.2 Database

The database cut-off date was 31 March 2020. The Mineral Resource estimation was undertaken in Leapfrog Geo using the Leapfrog Edge module. The database for this Mineral Resource estimate consisted of 972 drill holes totalling 355,593 m (Table 14-1); the current Mineral Resource estimate itself is based on 750 drill holes that have been drilled since 1993, a total of 339,234 m. A surface drill hole plan is shown in Appendix B.

Channel samples from the development mining areas were used to guide interpretation of the mineralized vein wireframes. In all, 2,472 chip sample strings, amounting to 13,196 individual samples, are present in the chip sample database, of which 2,454 individual samples were used for the Mineral Resource estimate.

Table 14-1: Sugar Zone Mine Drill-Hole Database Summary

Year Drilled	No. of Holes	Metres Drilled
2019	28	15,236
2018	256	112,507
2017	279	142,737
2016	53	19,957
2012–2015	151	22,304
2010–2011	73	16,323
1983–2009	132	26,529
Total	**972**	**355,593**

All drill-hole survey and assay values are expressed in the International System of Units (i.e., metric), with grid coordinates in the NAD 83, Zone 16N Universal Transverse Mercator (UTM) system.

14.3 Domain Interpretation

Interpreted wireframes of the mineralized veins were created for the Sugar Zone (Upper, Lower, and two Footwall Zones), Middle Zone (Middle Zone, three Footwall Zones, one Hanging Wall Zone) and the Wolf Zone (2 zones). Figure 14-1 shows a 3-D perspective view of the vein packages. The tops of the interpretations are clipped against topography and existing mine workings are shown for reference.

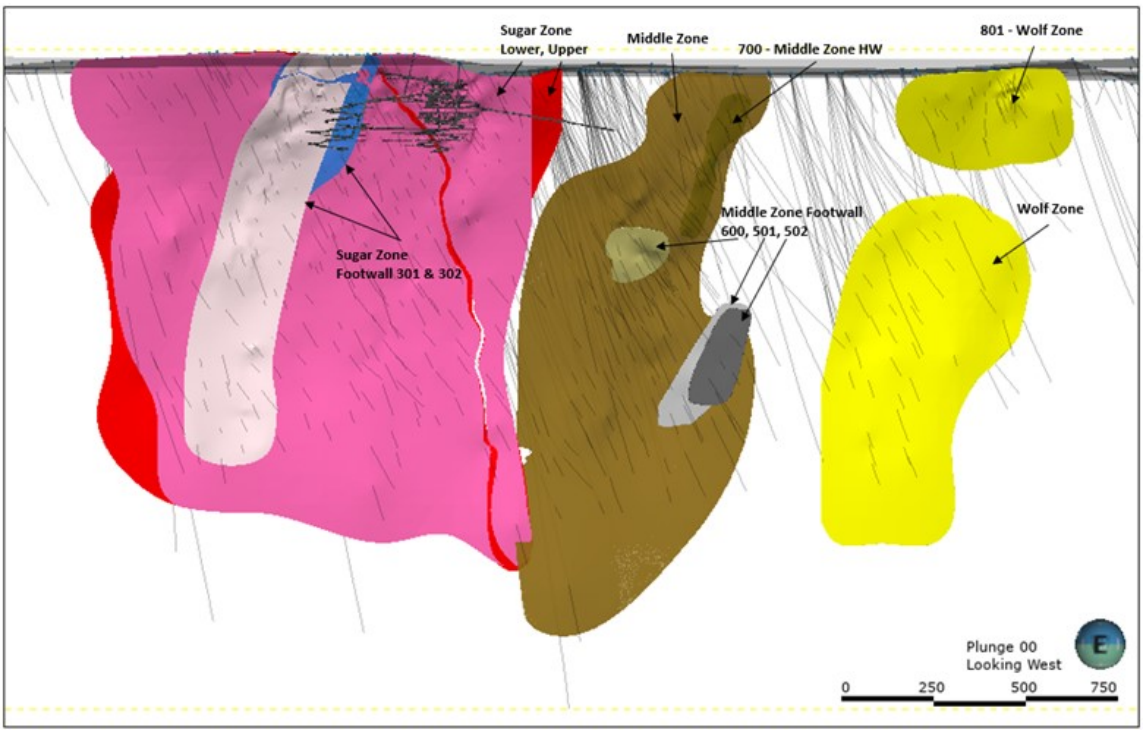


Figure 14-1: Interpreted Vein Wireframes 3-D Perspective Looking West

The wireframes were created from successive interpretations based on drill-core logging and underground mapping. For the mineralized domain interpretations, the contact between mineralized and non-mineralized material is very sharp and narrow, and a strict assay cut-off is not generally used for domain interpretations. The vein modelling tool in Leapfrog Geo (Version 5.1) was used to create the 3-D mineralized wireframes. In some cases, low-grade mineralization was included to maintain zonal continuity. Where the sectional interpretations lay close to drill intercepts or channel samples, interpretations were snapped to the drill hole or channel intercepts in three dimensions. The resulting wireframes were used as constraining boundaries during the Mineral Resource estimation process for block coding, statistical analysis, and compositing limits.

The topographic surface was created by Harte Gold from drill-hole collar points, and compared to the lidar survey data in the area of interest.

Underground mine infrastructure (ramps and levels) were surveyed using a Leica TS12 total station. Stopes were surveyed using an underground cavity monitoring system (CMS) using a Leica P20 high-definition scanner. The surveyed point cloud data were used to generate a 3-D representation of extracted stopes. The combined drift survey solids and stope CMS were used to deplete what had been extracted from the Mineral Resource estimate during drifting and mining, in both the bulk sample area and the commercially mined areas.

As shown in Table 14-2, a unique rock code was assigned for each mineralized vein wireframe in the Mineral Resource model.

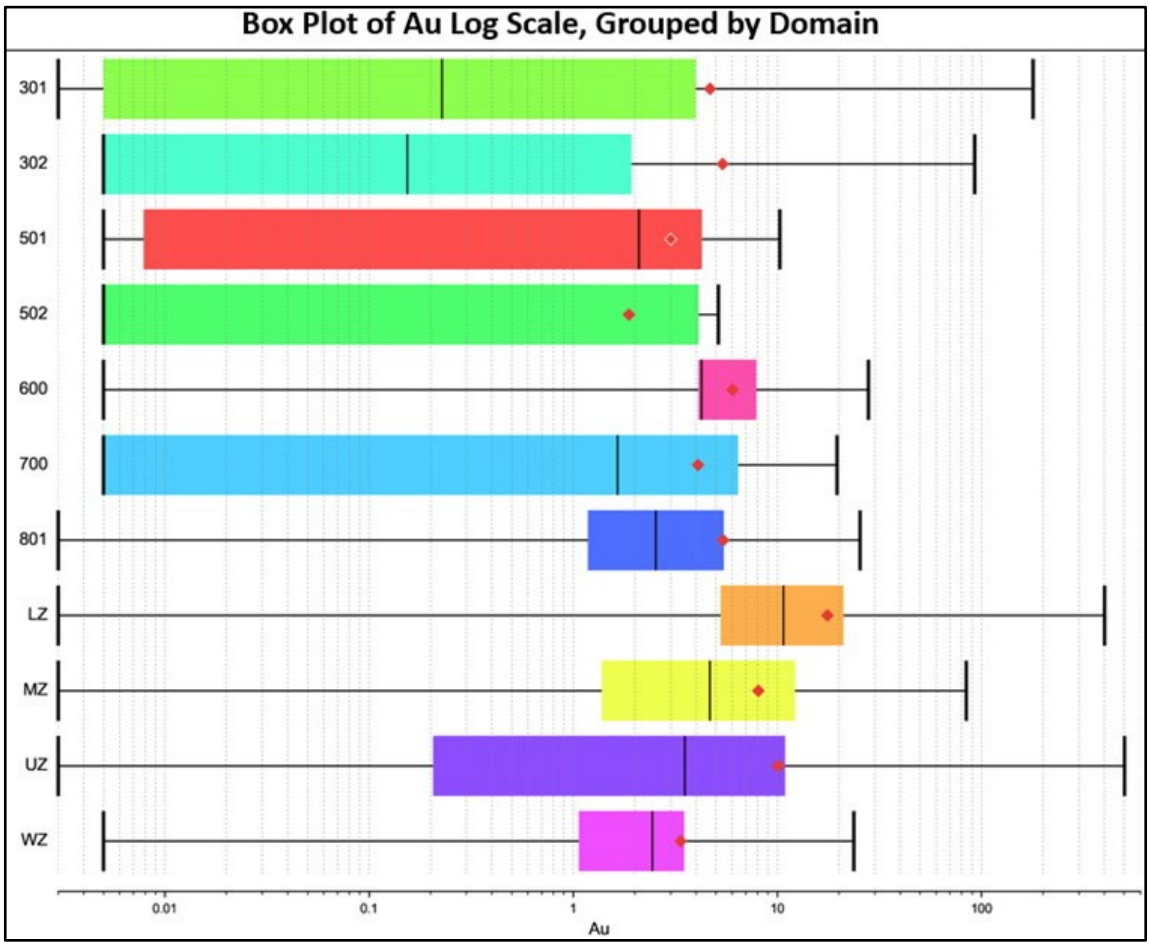
Table 14-2: Block Model Codes and Volume of Vein Wireframes

Domain	Code	Volume (m^3 '000s)
Sugar Zone Lower	LZ	1,250
Sugar Zone Upper	UZ	1,228
Sugar Zone FW1—301	301	148
Sugar Zone FW2—302	302	91
Middle Zone	MZ	1,047
Middle Zone FW1—501	501	66
Middle Zone FW2—502	502	88
Middle Zone FW3—600	600	56
Middle Zone HW—700	700	52
Wolf Zone—801	801	134
Wolf Zone	WZ	393

14.4 Exploratory Data Analysis and Compositing

The average sampling length is 0.5 m. Full vein-width composites were used. In previous Mineral Resource estimates 1.5 m, 1.25 m, and 1 m composites were used. Full-width composites were used to perform 2-D kriging, which is optimal for narrow vein deposits such as Sugar Zone, and gives a result that is a more realistic representation of actual mining selectivity. These full-width composites improved variogram modelling, and provide a realistic reflection of the underlying mineralization variability in relation to the thickness of the vein wireframes. Fewer than 50 composites were available in Domains 501, 502, 600, 700, 801, and WZ; for these, 2-D inverse distance squared (ID2) was performed instead of kriging. Figure 14-2 illustrates a set of box plots showing full-width composites by vein wireframe. Grade statistics are length weighted in all cases.

Compositing was commenced and halted on the interpreted wireframe boundaries. Once these full-width grade composites were created, full-width true thickness composites were generated by attributing the true modelled thickness to each of these grade composites. The true thickness was attributed in Leapfrog Geo (Version 5.1) using the Evaluations function, under Points. Visual and statistical checks were performed on the true thickness and the grade full-width composites.



Domain	Attribute	Count	Mean	Standard Deviation	Coefficient of Variation	Variance	Minimum	Lower Quartile	Median	Upper Quartile	Maximum
Sugar Zone Lower (LZ)	Au	2460	17.62	23.77	1.35	564.91	0.003	5.28	10.72	21.11	401.29
Sugar Zone Upper (UZ)	Au	850	10.12	18.96	1.87	359.42	0.003	0.21	3.52	10.92	502.18
Sugar Zone FW1—301	Au	118	4.68	16.22	3.46	263.05	0.003	0.01	0.23	3.98	179.00
Sugar Zone FW2—302	Au	79	5.40	15.94	2.95	254.01	0.01	0.01	0.15	1.93	92.60
Middle Zone (MZ)	Au	149	8.07	11.08	1.37	122.75	0.003	1.38	4.68	12.20	84.29
Middle Zone FW1—501	Au	8	3.01	3.88	1.29	15.02	0.01	0.01	2.10	4.27	10.27
Middle Zone FW2—502	Au	6	1.87	2.29	1.22	5.26	0.01	0.01	0.01	4.12	5.14
Middle Zone FW3—600	Au	10	6.03	5.95	0.99	35.34	0.01	4.10	4.26	7.89	27.94
Middle Zone HW—700	Au	18	4.08	5.47	1.34	29.95	0.01	0.01	1.65	6.42	19.60
Wolf Zone—801	Au	48	5.44	6.94	1.28	48.19	0,003	1.18	2.54	5.48	25.38
Wolf Zone (WZ)	Au	39	3.37	4.13	1.23	17.02	0.01	1.06	2.44	3.50	23.70

Figure 14-2: Composites Boxplots and Statistics

14.5 Outlier Analysis

Outlier analysis and grade capping were investigated on the assays and the full-width composite values in the database, within the constraining wireframes to ensure that the possible influence of erratic high values did not bias the estimation. In the case of the Sugar Zone deposit, 5% to 10% of the metal is risk-adjusted from the grade cap applied on the full-width composites. The

Sugar Zone has been greatly de-risked by over 18 months of commercial mining production, reconciliation, and channel sampling.

Grade capping was performed on the full-width composites, and several analytical methods were applied, and broadly align. The following methods were used: probability plot analysis; disintegration of the tail of the histogram; decile analysis; decrease of the coefficient of variation; and, metal loss. Various statistical plots used in the work are included in Appendix C. Final grade capping levels are shown in Table 14-3.

Table 14-3: Full Width Composite Grade Capping Levels

Domain	Grade Cap Au (g/t)
Sugar Zone Lower (LZ)	85
Sugar Zone Upper (UZ)	70
Sugar Zone FW1—301	60
Sugar Zone FW2—302	60
Middle Zone (MZ)	45
Middle Zone FW1—501	20
Middle Zone FW2—502	20
Middle Zone FW3—600	20
Middle Zone HW—700	20
Wolf Zone—801	20
Wolf Zone (WZ)	40

14.6 Spatial Data Analysis and Variography

Two variograms were built for each domain: one for the full-width composites of the accumulation (grade-thickness), and one for the true thickness of the full-width composites. The nugget effect for each domain was evaluated with the use of an omnidirectional variogram within the plane of the vein.

The Lower Zone variograms in the best direction of continuity (dip 70°, dip direction 235°) show correlation out to 125 m. This direction agrees with the southwest-plunging L2 lineation from the structural study work to date. The Lower Zone variograms were also used for Domain 301 and 302.

For the Upper Zone and Middle Zone, a normal score variogram was calculated and then back transformed. In the best direction of continuity (dip 70°, dip direction 235°) show correlation out to 300 m.

The variography study highlighted the sensitivity to global and local changes in structural orientation of the vein packages. In particular, the overall dip changes quite substantially with elevation, and there are also local changes in orientation due to rolls and flexures. The need for using some form of unrolling for the estimation is noted and discussed in Section 14.8.

Selected variograms are included in Appendix D. Figure 14-3 shows the final variogram models used for the Lower Zone accumulation (grade thickness) estimation.

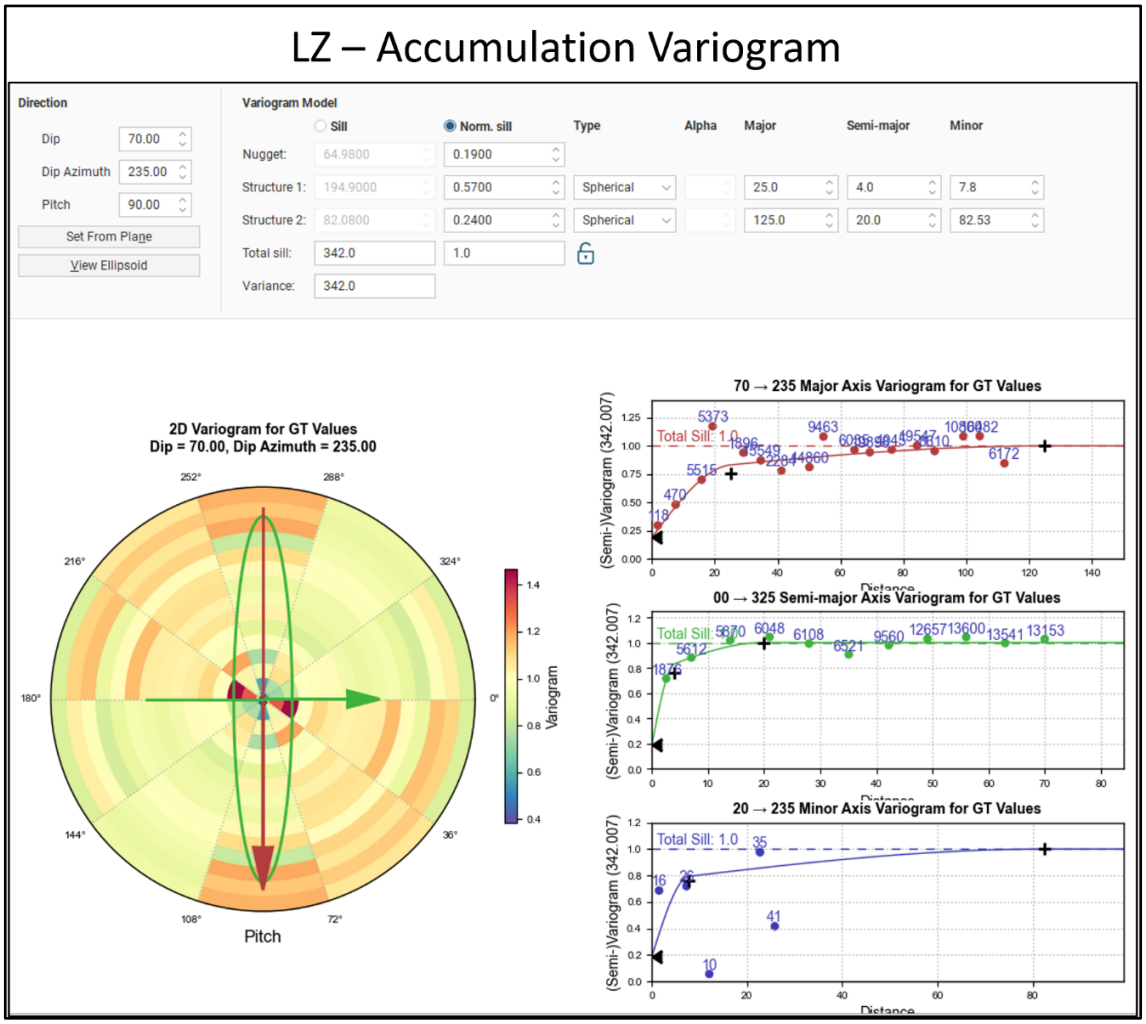


Figure 14-3: Full Width Composites—Variogram Model

14.7 Bulk Density

The database has been updated with 179 bulk density measurements taken during the Lower Zone bulk sample mining, with values varying between 2.53 and 3.02 t/m^3, with a weighted average of 2.73 t/m^3. A uniform bulk density of 2.73 t/m^3 was used for all domains for the Mineral Resource estimate.

14.8 Block Modelling

The Sugar Zone Mineral Resource Estimate block model was constructed using Leapfrog Geo Edge (Version 5.1) software and the block model origin and block size are tabulated in Figure 14-4.

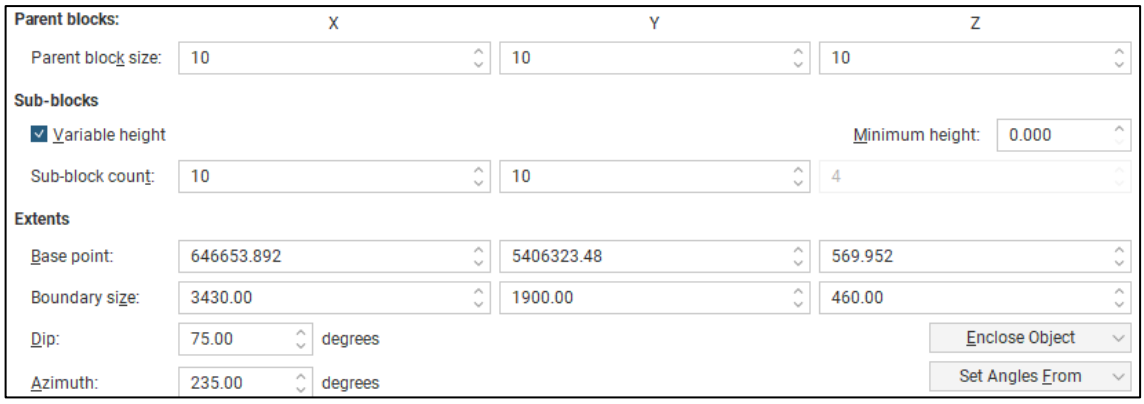


Figure 14-4: Block Model Definition—Leapfrog Geo Edge

Multi-pass block estimation was performed in each wireframe. In each case, only those composites flagged by the pertinent wireframe were used for estimation. The block model was rotated, and a variable height was used, so only one block was sub-blocked in the Z direction (across strike). A minimum height of 0.25 m was also tested and did not impact the estimation. In this case, since the block model is rotated, the X-axis represents the vertical, the Y-axis is in the strike length, and the Z-axis is across strike. Au estimation in all domains was performed using 2-D kriging or 2-D ID2.

- ID2 was used for Domains 501, 502, 600, 700, 801, and WZ
  - Less than 50 composites available for each of these domains
- Ordinary kriging (OK) was used for Domains 301, 302, LZ, UZ, and MZ
  - Domains 301 and 302 use the LZ variogram.

Additional estimation methods were used for validation, including OK capped, OK uncapped, nearest-neighbour (NN), and ID2.

The variable orientation option in the estimation section of Leapfrog Geo Edge was used. The variable orientation strategy or dynamic anisotropy search strategy was used for all estimations. This approach uses the following steps in each block:

- Start with a base search ellipse orientation (from the domain variogram)
- Subsequently modify the search ellipse orientation based on the surrounding modelled three-dimensional geometry of the vein.

Figure 14-5 illustrates the general approach to the use of dynamic anisotropy orientations. The orange lines within each block indicate the direction (down-plunge) of the search ellipsoids.

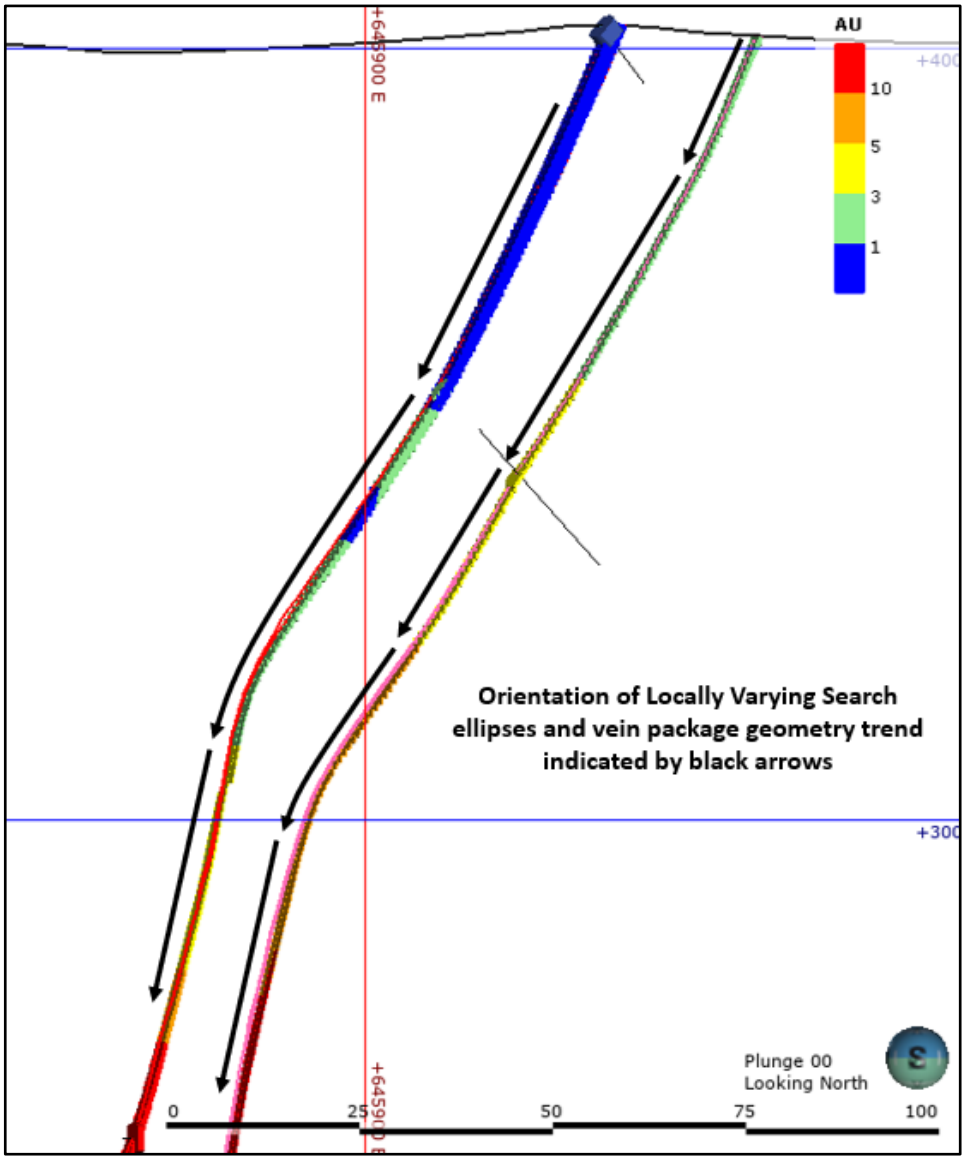


Figure 14-5: Locally Varying Search Ellipsoid Orientations Looking North

Grade blocks were interpolated using the parameters given in Table 14-4.

Table 14-4: Block Model Grade Interpolation Parameters

Pass	Ellipsoid			Search		Outlier Restriction	
	Maximum (m)	Intermediate (m)	Minimum (m)	Minimum No. Samples	Maximum No. Samples	Clamp–Distance (% of search)	Value Threshold (m*g/t)
A	15	10	25	4	40	20	25
0	15	10	25	3	7	-	-
1	50	25	25	3	7	-	-
2	100	60	50	2	6	-	-
3	200	120	80	2	5	-	-

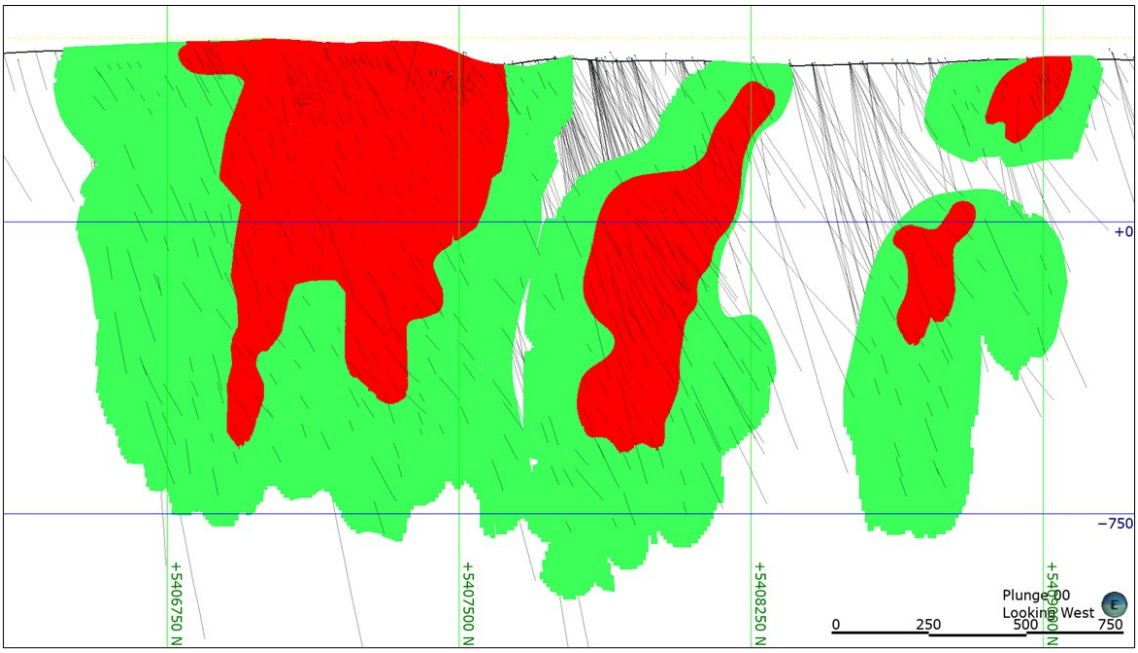
Pass A was interpolated with samples of underground channels and drill holes, while the Passes 0, 1, 2, and 3 were interpolated with drill-hole samples only. Pass A uses a clamp on the high-grade samples to restrict the influence and spread of the high-grade channel samples in the commercial mining areas. Selected cross-sections and plans of the gold grade blocks are presented in Appendix E.

14.9 Mineral Resource Estimate Classification

In the QP's opinion, the drilling, assaying, and exploration work on the Sugar Zone deposit support this Mineral Resource estimate, and are sufficient to indicate a reasonable potential for economic extraction, thus qualifying it as a Mineral Resource under the *CIM Definition Standards for Mineral Resources & Mineral Reserves* (CIM Definition Standards). The Mineral Resource was classified as Indicated and Inferred based on the following criteria:

- Indicated (rclass == 2)
  - Nominal drill spacing average 50 m
  - Supported by 125 m range on variogram down-plunge and reconciliation
  - De-risked by commercial mining and channel sampling
  - Minimum of three drill holes used to estimate any block.
- Inferred (rclass == 3)
  - Nominal drill spacing average 100 m
  - Predicated on strong geologic interpretation
  - Minimum of two drill holes used in all block estimates
  - Less than 5% of blocks in extrapolation.

Additional smoothing and removal of small pods were applied to finalize the classification to make it amenable for underground mine planning. Figure 14-6 illustrates a longitudinal projection showing the Mineral Resource estimate classification. Selected cross-sections of the Mineral Resource estimate classifications are presented in Appendix F.



Note: Red = Indicated Mineral Resource; Green = Inferred Mineral Resource.

Figure 14-6: Longitudinal Projection Looking West Showing Mineral Resource Classification

14.10 Resource Cut-off Grade Estimate

The Mineral Resource estimate was derived by applying Au grade cut-off values to the block model and reporting the resulting tonnes and grade for potentially underground mineable areas. The following parameters were used to estimate the cut-off grade.

Table 14-5: Au Cut-Off Grade Calculation

Parameter	Value
Au Price	US$1,600/oz
Exchange Rate	US$0.75:C$1.00
Mining Cost/Tonne Mined	$110/t
Processing Cost/Tonne Processed	$35/t
G&A/Tonne Processed	$58/t
Au Process Recovery	94.0%
Payable Gold	98%
Royalty	3.5%
Treatment/Recovery Cost	$50/oz

A grade-thickness cut-off was used to ensure that the material reported above the cut-off grade is minable and respects the grade thickness equivalent of a minimum mining width of 1.8 m.

The estimate results in a cut-off grade of 3.0 g/t and a grade-thickness cut-off grade of 5.4 m*g/t.

14.11 Mineral Resource Estimate

The resulting Mineral Resource estimate is tabulated in Table 14-5. The QP considers the mineralization at the Sugar Zone deposit, including the Middle and Wolf Zones, to be potentially amenable to underground extraction.

Table 14-6: Mineral Resource Estimate at 3.0 g/t Au Cut-off Grade and 5.4 m*g/t Grade-Thickness Cut-off

Classification	Zone	Tonnes (t '000s)	Au (g/t)	Au (oz '000s)
Indicated	Sugar (North and South)	1,315	15.53	657
	Middle	1,326	8.95	381
	Wolf	162	6.13	32
	Total	**2,803**	**11.87**	**1,070**
Inferred	Sugar (North and South)	891	10.98	314
	Middle	844	8.21	223
	Wolf	132	7.04	29
	Total	**1,866**	**9.45**	**567**

Notes: The stated Mineral Resources comply with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101), and are classified in accordance with the CIM Definition Standards.
Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.
The Mineral Resource is inclusive of the Mineral Reserves.
The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
The Mineral Resource estimate was prepared by Mr. Vincent Cardin-Tremblay (P.Geo.), former VP Geological Services for Harte Gold. Mr. Cardin-Tremblay is a QP as defined by NI 43-101.
Mineral Resources have been estimated as of 30 September 2020 using a gold price assumption of US$1,600 per ounce Au.
A resource cut-off grade of 3.0 g/t Au has been estimated based on OPEX projections and applicable metallurgical recovery. The cut-off grade was used in combination with a minimum mining width factor of 1.8 m to define the resource.
Totals may not add due to rounding.

Mineral Resources are sensitive to the selection of a reporting gold cut-off grade. The sensitivities to the gold cut-off grade are demonstrated in Table 14-7.

Table 14-7: Sugar Zone Mineral Resource Estimate Sensitivity

Classification	Cut-off Grade Au (g/t)	Grade-Thickness Cut-off (m*g/t)	Tonnes ('000s)	Grade Au (g/t)	Contained Au (oz '000s)
Indicated	4.50	8.1	2,084	13.81	925
	4.00	7.2	2,297	13.19	974
	3.50	6.3	2,520	12.60	1,021
	3.00	5.4	2,803	11.87	1,070
	2.50	4.5	3,097	11.24	1,119
	2.00	3.6	3,366	10.71	1,159
Inferred	4.50	8.1	1,159	11.14	415
	4.00	7.2	1,349	10.60	459
	3.50	6.3	1,546	10.10	502
	3.00	5.4	1,866	9.45	567
	2.50	4.5	2,156	8.91	618
	2.00	3.6	2,522	8.31	674

14.12 Confirmation of Mineral Resource Estimate

The block model was validated using several industry standard methods including:

- Visual validation comparing block estimates to sampling on sections and plans
- Global comparison to de-clustered distributions; this involves comparing mean gold grades between estimated blocks and the NN estimate
- Comparison to previous resource estimates
- Comparison to OK, ID2, and NN estimations
- Global swath plots comparing estimates to composites on a more local scale, with elevation, northing, and easting
- Evaluation of the effect of grade capping.

By visually checking on sections and plans, the block grade distributions are somewhat smoothed, especially in Inferred blocks; consequently, the model will not likely provide accurate local Inferred grade estimates. This issue is caused by the highly variable nature of Au grades, wide drill-hole spacing, and the interpolation method. Infill drilling is warranted to better model local variations of gold grade. In the QP's opinion, the global tonnage and grade estimates are reasonable.

The QP considers the Mineral Resource estimate to be reasonable and in conformity with the generally accepted *CIM Estimation of Mineral Resources & Mineral Reserves Best Practices Guidelines* (CIM, 2019).

Local validation within domains using swath plots was applied for groups of Indicated + Inferred blocks. Validations were confirmatory in all cases. Figure 14-9 to Figure 14-9 show an example comparison for Indicated + Inferred blocks in the Lower Zone, Upper Zone, and Middle Zone. The comparison is good, showing some local variation of the block estimates following the trends of the de-clustered composite values (NN). There is also adequate evidence of smoothing in the block estimates versus the point estimates (de-clustered composite values in the NN).

Lower Zone

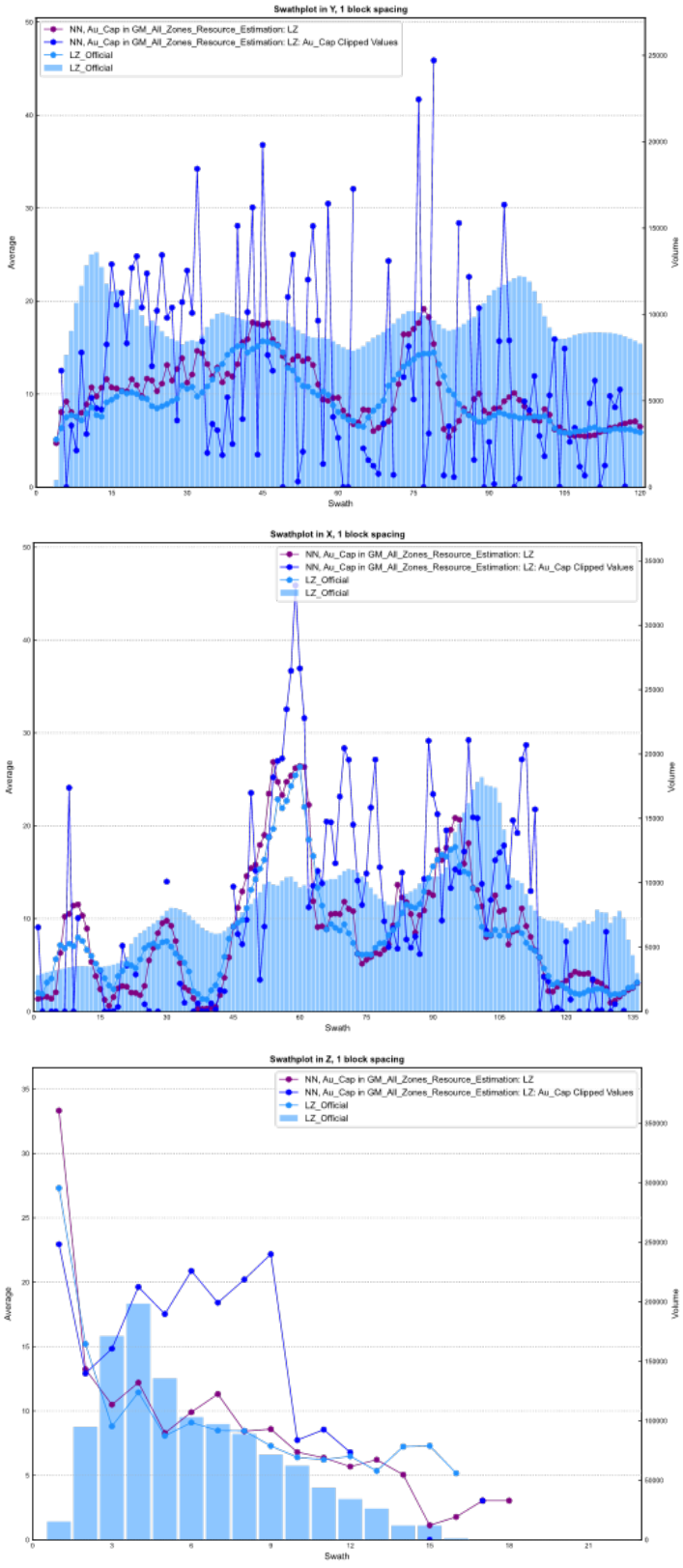


Figure 14-7: Swath Plots for Lower Zone Indicated + Inferred Blocks

Upper Zone

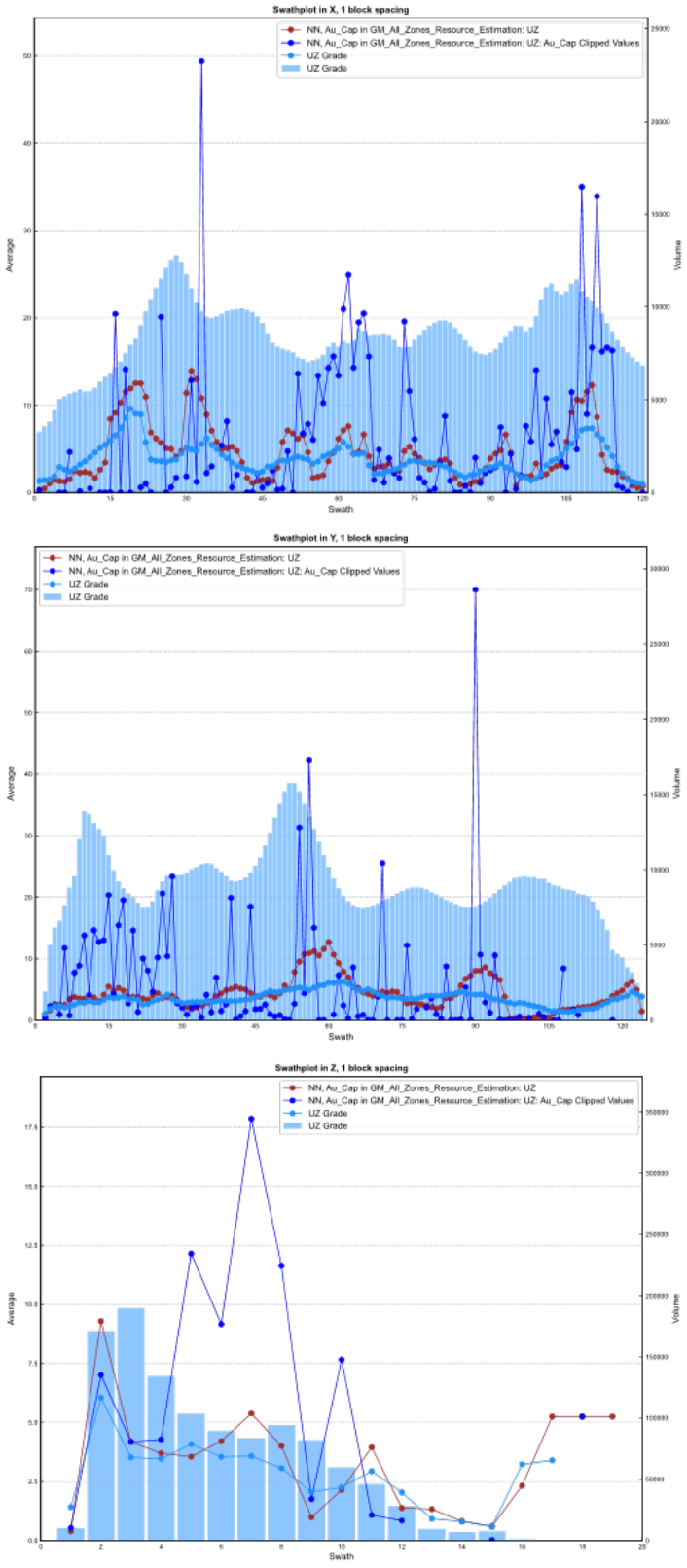
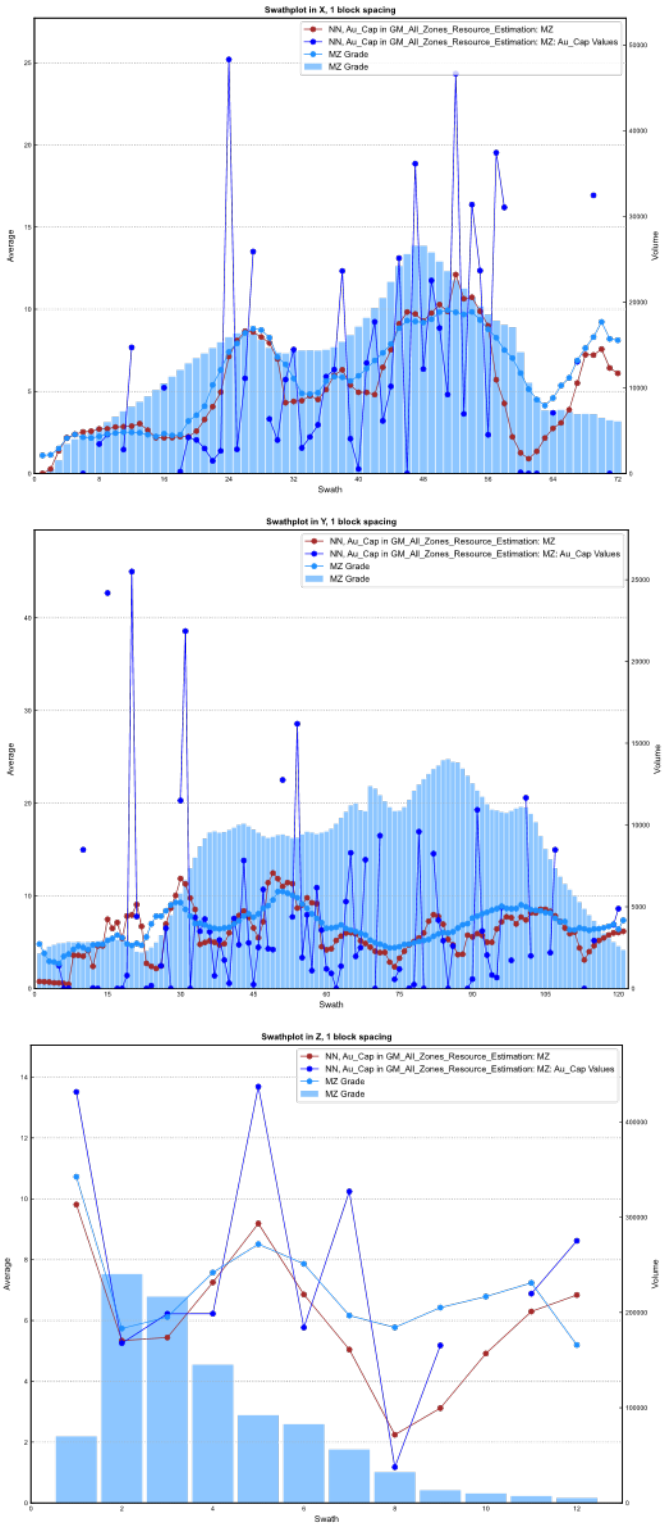


Figure 14-8: Swath Plots for Upper Zone Indicated + Inferred Blocks

Middle Zone



Source: Harte Gold, 2020.

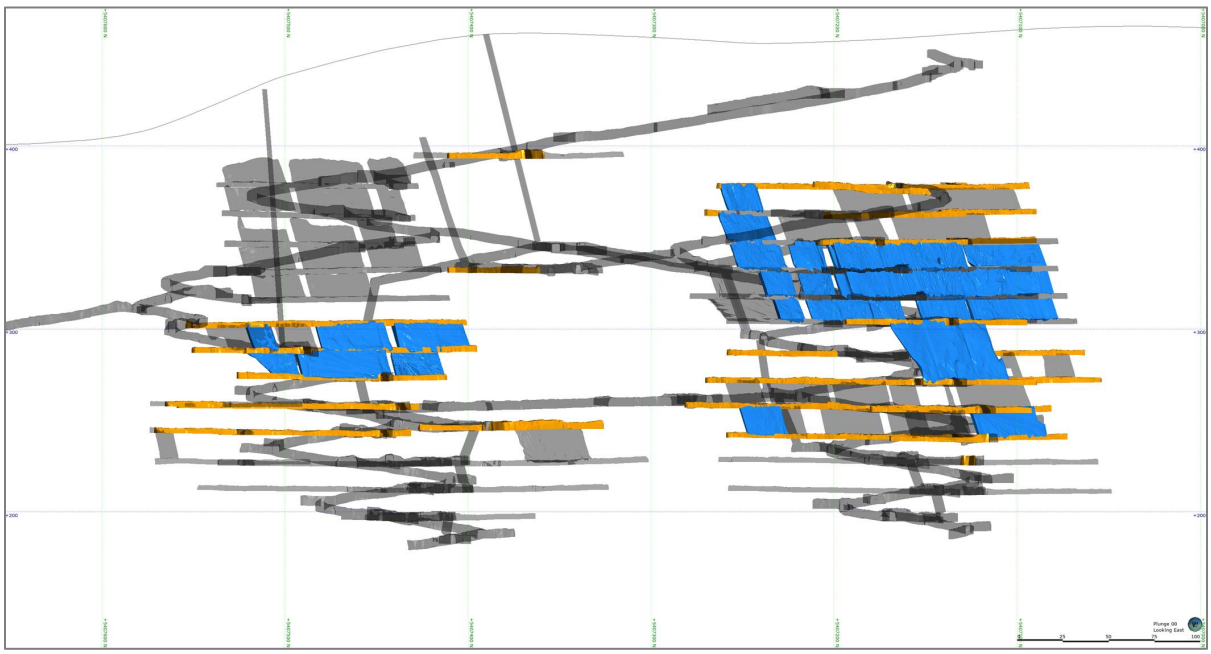
Note: Since the block model is rotated, the x axis represents the vertical, the y axis is in the strike length and the z axis is across strike.

Figure 14-9: Swath Plots for Middle Zone Indicated + Inferred Blocks

14.12.1 Reconciliation

The Sugar Zone Mine entered commercial production in 2019. At the time of this Technical Report, over 18 months of commercial production data are available to reconcile with the Mineral Resource model.

In 2019, 2,798 m of development and 27 stopes were mined out (Figure 14-10).



Note: Blue indicates production blocks extracted in 2019; orange indicates sills extracted in 2019; gray represents previously mined workings to the end of September 2020.

Figure 14-10: 2019 Development and Mined Out Stopes (Looking East)

A total of 175,173 t (dry) at 4.85 g/t Au was processed at the Sugar Zone Mine process plant in 2019, a yield 27,851 oz Au. The total tonnage was a combination of production and development, as shown in Table 14-7. A low-grade stockpile from the bulk test and pre-production development contributed 24,886 tonnes at 2.70 g/t Au. When combined with the available underground and surface stockpiles (1,155 tonnes at 3.93 g/t Au), a total of 26,041 tonnes at 2.75 g/t of stockpiled material was available for processing. The combined total of processed and stockpiled mineralized material amounted to 201,214 tonnes at 4.57 g/t Au.

Table 14-8: Summary of 2019 Production and Development

	Reconciled to Mill		Stockpile		Total	
	Tonnes (dry)	Grade (g/t Au)	Tonnes[1] (dry)	Grade[2] (g/t Au)	Tonnes (dry)	Grade (g/t Au)
Production	103,212	5.27	482	5.30	103,694	5.27
Development	71,961	4.33	212	0.86	72,173	4.23
Surface Stock			461	3.91	461	3.91
Low Grade Stock			24,886	2.70	24,886	2.70
Total	**175,173**	**4.85**	**26,041**	**2.75**	**201,214**	**4.57**

Notes: [1] Stockpiled tonnes based on truck counts and loader measured tonnes.
[2] Au Grade based on mill reconciled numbers.

The achieved Au grade of the stope and development was reconciled to the current Mineral Resource estimate model. Not all material that was mined was processed; 2% of the ore had not gone through the process plant by the end of 2019. Reconciliation comparisons were done on the reconciled mill numbers.

For the development ore tonnes, the current Mineral Resource estimate model was clipped using drift outlines based on the extent of the development, to provide tonnes of grade within the Mineral Resource shape. The in-situ tonnes and grade were diluted by waste tonnes at zero grade for material outside the Mineral Resource outlines. Table 14-9 summarizes the block model-diluted estimated tonnes and grade versus the process-plant reconciled numbers. The block model estimated grade of 4.24 g/t Au compared to a processed grade of 4.33 g/t Au. The block model estimated 71,653 tonnes compared to 71,961 tonnes processed.

Similarly, the stope outlines based on the CMS were used to clip the current Mineral Resource estimate model to provide the in-situ tonnes and grade of the stope production. This in situ tonnage was diluted at zero grade for material outside the Mineral Resource outlines. Table 14-9 summarizes the block-model diluted estimated tonnes and grade versus the process-plant reconciled numbers. The block model estimated grade of 5.16 g/t Au compared to a processed grade of 5.27 g/t Au. The block model estimated 102,722 tonnes compared to 103,212 tonnes processed.

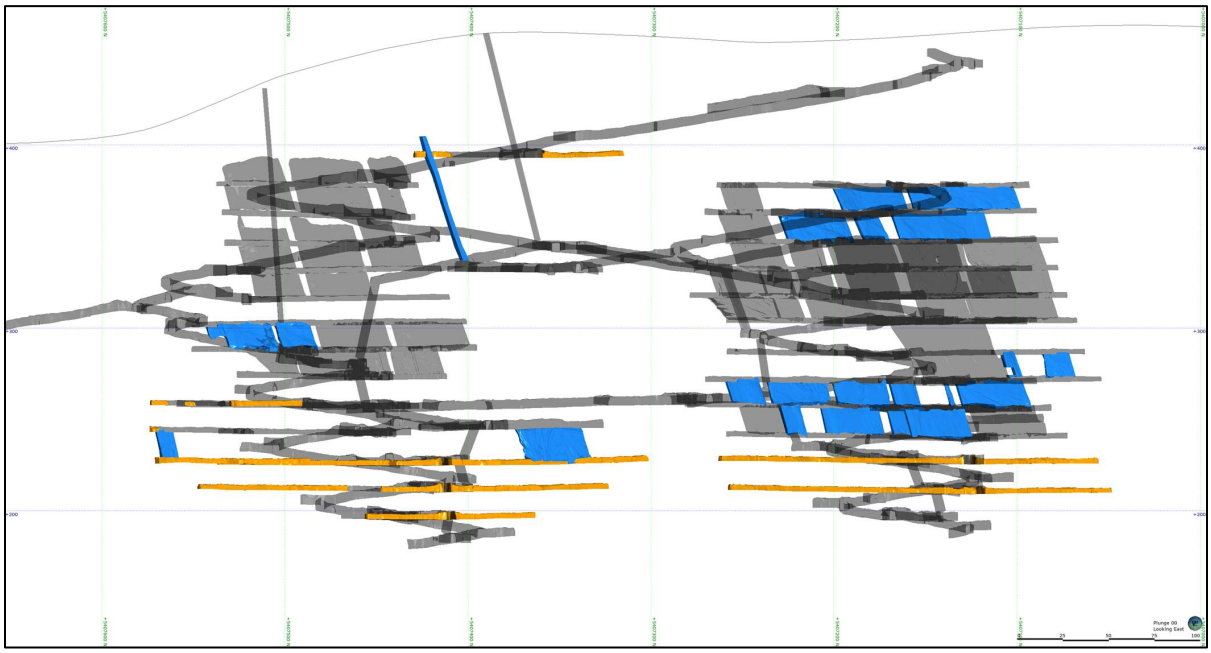
Table 14-9: 2019 Reconciled Tonnes and Grade to Mineral Resource Model

Item	Mill Reconciled Numbers		Mineral Resource		Reconciliation	
	Tonnes (dry)	Au (g/t)	Tonnes (dry)	Au (g/t)	Tonnes (%)	Grade (%)
Development	71,961	4.33	71,653	4.24	99.5	97.9
Production	103,212	5.27	102,772	5.16	99.5	97.9
Total	**175,173**	**4.85**	**174,375**	**4.78**	**99.5**	**97.9**

Overall, the combined development and production from the block model was estimated at a grade of 4.78 g/t Au compared to a processed grade of 4.88 g/t Au, an underestimation of the grade by

0.1 g/t Au, or 2.1%. The block model estimated 174,375 tonnes compared to 175,173 tonnes processed, an underestimation of 798 tonnes, or 0.5%.

In the first nine months of 2020, 1,307 m of development and 22 stopes were mined fully or partially (Figure 14-11).



Note: Blue indicates production blocks; orange indicates sills extracted; gray represents previously mined workings to the end of September 2020.

Figure 14-11: 2020 Development and Mined Out Stopes (Looking East)

A total of 85,766 tonnes (dry) at 5.44 g/t Au (15,005 oz) was processed at the Sugar Zone Mine process plant in the first nine months of 2020. The total tonnage was a combination of production and development as shown in Table 14-10. A combination of surface and underground stockpiles contributed 3,109 tonnes at 8.21 g/t Au. The combined total of processed and stockpiled mineralized material amounted to 88,874 tonnes at 5.54 g/t Au.

The achieved gold grade of the stope and development was reconciled to the current Mineral Resource estimate model. Not all material that was mined was processed; 3% of the ore did not go through the process plant prior to 30 September 2020. Reconciliation comparisons were done on the reconciled mill numbers.

Table 14-10: Summary of 2020 Production and Development

	Reconciled to Mill		Stockpile		Total	
	Tonnes (dry)	Grade (g/t Au)	Tonnes[1] (dry)	Grade[2] (g/t Au)	Tonnes (dry)	Grade (g/t Au)
Production	56,471	6.20	635	9.42	57,106	6.24
Development	29,295	3.98	190	12.28	29,485	4.03
Surface Stock	-	-	2,284	7.54	2,284	7.54
Total	**85,766**	**5.44**	**3,109**	**8.21**	**88,875**	**5.54**

Notes: [1] Stockpiled tonnes based on truck counts and loader measured tonnes.
[2] Au Grade based on mill reconciled numbers.

For the development ore tonnes, the current Mineral Resource estimate model was clipped using drift outlines based on the extent of the development, to provide tonnes of grade within the Mineral Resource shape. The in-situ tonnes and grade were diluted by waste tonnes at zero grade for material outside the Mineral Resource outlines. Table 14-11 summarizes the block-model diluted estimated tonnes and grade versus the process-plant reconciled numbers for development ore. The block model estimated grade of 3.72 g/t Au compared to a processed grade of 3.98 g/t Au. The block model estimated tonnage of 27,759 tonnes compared to 29,295 tonnes processed.

Similarly, the stope outlines based on the CMS were used to clip the current Mineral Resource estimate model to provide the in-situ tonnes and grade of the stope production. This in situ tonnage was diluted at zero grade for material outside the Mineral Resource outlines. Table 14-11 summarizes the block-model diluted estimated tonnes and grade versus the process-plant reconciled numbers for stoping production ore. The block model estimated grade of 5.79 g/t Au compared to a processed grade of 6.20 g/t Au. The block model estimated 53,395 tonnes compared to 56,471 tonnes processed.

Overall, the combined development and production from the block model was estimated at a grade of 5.08 g/t Au compared to a processed grade of 5.44 g/t Au, an underestimation of 0.36 g/t Au, or 6.7%. The block model estimated 81,154 tonnes compared to 85,766 tonnes processed, an underestimation of 4,612 tonnes, or 5.4%.

Table 14-11: 2020 Reconciled Tonnes and Grade to Mineral Resource Model

	Mill Reconciled Numbers		Mineral Resource		Reconciliation	
Item	Tonnes (dry)	Au (g/t)	Tonnes (dry)	Au (g/t)	Tonnes (%)	Grade (%)
Development	29,295	3.98	27,759	3.72	94.7	93.4
Production	56,471	6.20	53,395	5.79	94.5	93.4
Total	**85,766**	**5.44**	**81,154**	**5.08**	**94.6**	**93.4**

Finally, the combined development and production of 2019 and 2020 from the block model was estimated at a grade of 4.88 g/t Au compared to a processed grade of 5.07 g/t Au, resulting in an

underestimation of the grade by 0.19 g/t Au, or 3.7% (Table 14-12). T The block-model estimated 255,529 tonnes compared to 260,939 tonnes processed, an underestimation of 5,410 tonnes, or 2.1%.

Table 14-12: 2019 to End of September 2020—Reconciled Tonnes and Grade to Mineral Resource Model

Item	Mill Reconciled Numbers		Mineral Resource		Reconciliation	
	Tonnes (dry)	Au (g/t)	Tonnes (dry)	Au (g/t)	Tonnes (%)	Grade (%)
Development	101,256	4.23	99,412	4.09	98.2	96.8
Production	159,683	5.60	156,117	5.38	97.8	96.0
Total	**260,939**	**5.07**	**255,529**	**4.88**	**97.9**	**96.3**

Reconciliation is an ongoing process at the Sugar Zone Mine, and different improvement plans are currently being driven to improve the reconciliations. Some of these improvement plans are:

- Refine lithological model
- Specific gravity measurements to improve density model
- QA/QC evaluation of the CMS
- Grade-control drilling program to improve grade forecast
- QA/QC evaluation of stockpile estimation.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

Current operations use a sublevel open-stoping method with unconsolidated waste backfill. Paste backfill will be introduced in 2023 to replace the unconsolidated waste. The mining sequence progresses down the orebody with sublevels spaced 15 to 20 m vertically.

The mineralization typically comprises tabular, narrow veins, varying in thickness from 0.2 to 4.4 m, and dipping at around 70° to the southwest.

In the North and South Zones, the average true width of the ore is 1.1 m, with a range of 0.2 to 3.0 m, and the average stope design true width is 2.0 m, with a range of 1.5 to 3.1 m. A mining block consists of three 15 m high sublevels. Within each 45 m high block, a bottom-up sequence is employed, retreating each sublevel back towards the centrally located main access, backfilling the stopes and continuing up to the next sublevel. A 2 m high sill pillar is left between the top stope of each block and the block above. The first two lifts are drilled with down holes, and the last with up holes to the bottom of the sill pillar.

In the Middle Zone the average true width of the ore is 2.2 m, with a range of 0.3 to 4.4 m, and the average stope design true width is 2.4 m, with a range of 1.6 to 5.3 m. The same mining sequence strategy will be employed as in the North and South Zones, and due to the wider average width of the stopes, the sublevel spacing will be increased to 20 m.

Figure 15-1 shows a long section of the mine looking west. Each mining zone is accessed by a FW ramp. Further details of the mining method, sequence, and schedule are provided in Section 16 of this Technical Report.

15.2 Dilution and Recovery

Planned dilution is accommodated in the stope optimization process by specifying a minimum horizontal stope width of 2 m. This equates to approximately 1.9 m true width for the average 70° dipping stopes and down to 1.5 m for shallow dipping stopes.

Unplanned dilution is incorporated using a calculated factor to apply a minimum of 0.5 m dilution to all stopes (0.25 FW and 0.25 m HW).

Planned dilution is applied in Deswik Stope Optimizer (DSO) using a minimum horizontal stope width criterion of 2 m.

- The minimum true width of a 50° dip stope (minimum angle observed) would be 1.5 m.
- The minimum true width of a 70° stope (average angle observed) would be 1.9 m.

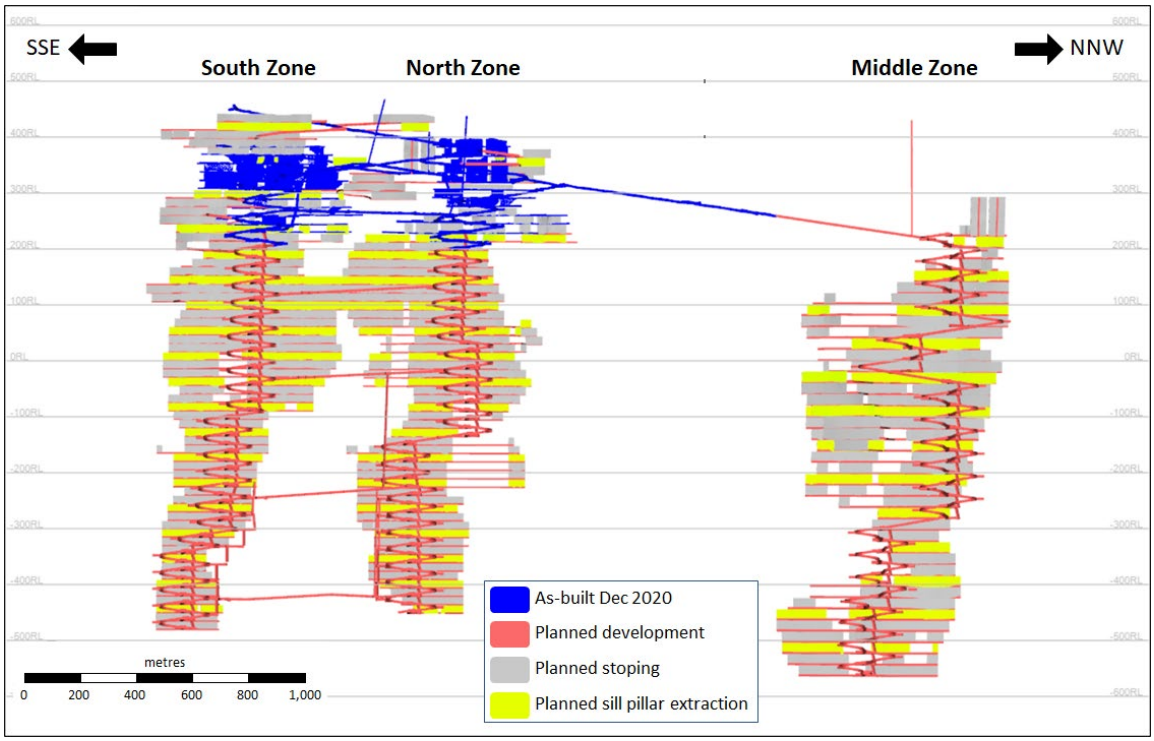


Figure 15-1: Long Section of the Mine Showing Stoping and Waste Development

Unplanned dilution is then applied as a factor in the design process as follows:

- When a DSO optimized stope shape was generated at the minimum mining width of 2 m horizontal:
- If vein width is ≤1.5 m, add unplanned dilution to reach 2 m minimum mining width. In this case the smallest possible DSO stope width would be 1.6 m, and 40 cm of dilution would be added to bring the width to 2.0 m.
- If vein width is 1.5 to 2.0 m add 0.5 m unplanned dilution to the average vein width. This would result in stopes between 2.0 and 2.5 m final width.
- When a DSO optimized stope shape was generated greater than the minimum mining width of 2m horizontal or vein width >2.0 m:
- Add 0.5 m unplanned dilution to the stope width.

Mining recovery was applied as follows:

- Non-sill pillar stopes—95% recovery
- Sill pillar stopes—85% to account for 2 m sill pillar left intact after uppers mining on final lift
- Sill Development—100%.

15.3 Operating Cost Estimates

The original OPEX estimates used for calculating cut-off grade for the stope optimization process were determined based on historical performance factored for expected changes to the production rate and operating methodology. The costs used in the final determination of the cut-off grade and presented in Section 15.4 are estimated using a zero-based methodology in all areas. The zero-based estimation validated the original factored historical costs and confirms the validity of the cut-off grade estimate used in the feasibility study.

15.4 Cut-off Grade

Table 15-1 presents the break-even and marginal cut-off grade estimates used in the stope optimization process. The estimate is prepared based on production and costs during the steady-state phase of the mine producing 1,200 t/d between 2023 and 2027.

Table 15-1: Break-Even and Incremental Cut-off Grade Estimates

			Unit	Break-Even Cut-off Grade	Marginal Cut-off Grade	Incremental Dev. Cut-off Grade
Costs (2023–2027)	Operating Cost	Mining	$ '000s	278,399	236,639	
		Processing	$ '000s	73,332	73,332	73,332
		G&A	$ '000s	94,362	94,362	94,362
	Sustaining Capital	UG cap. dev.	$ '000s	127,368		
Ore processed (2023–2027)			t '000s	2,146	2,146	2,146
Unit Costs	Operating Cost	Mining	$/t	130	110	0
		Processing	$/t	34	34	34
		G&A	$/t	44	44	44
	Sustaining Capital	UG dev.	$/t	59		
	Total		$/t	**267**	**188**	**78**
Gold Price			US$/oz	1,450	1,450	1,450
FOREX			US:C$	0.76	0.76	0.76
Gold Price			$/oz	1,908	1,908	1,908
Process Recovery		94.4%	$/oz	(107)	(107)	(107)
Payable Gold		98.3%	$/oz	(32)	(32)	(32)
TC/RC			$/oz	(50)	(50)	(50)
Royalty			$/oz	(67)	(67)	(67)
Net Revenue			$/oz	1,652	1,652	1,652
Net Revenue/Gram Au			$/g	53	53	53
Cut-off Grade			**g/t**	**5.0**	**3.5**	**1.5**

The marginal cut-off grade is derived by:

- Removing the sustaining capital development cost, as all sublevels will be accessed by ramps.
- Reducing the mining cost by 15% to account for sill development that is already in place to access high-grade ore.
- Applying full general and administrative (G&A) costs because marginal material replaces higher grade, rather than being mined in addition (due to mill capacity limitations).

The incremental development cut-off grade is derived by removing the sustaining capital development and mining costs, and accounting for processing and G&A costs.

15.5 Stope Optimization and Design

Table 15-2 shows the parameters that were used in the DSO evaluations.

Table 15-2: Deswik Stope Optimizer Parameters

Parameter	Unit	North and South Zones	Middle Zone
Level Height	m	15	20
Max. Stope Length	m	30–60	20–30
Minimum Width	m	2	2
Maximum Width	m	7	7
Stope Pillar	m	10	10
HW Dilution	m	0	0
FW Dilution	m	0	0
Default Seed Dip	°	70	70
Minimum Dip	°	50	50
Maximum Dip	°	130	130
Cut-off Grade	g/t	3.5	3.5

DSO was used to evaluate the resource block model and identify economic longhole stoping blocks above a 3.5 g/t Au marginal cut-off grade. A minimum horizontal stoping width of 2 m was specified to account for planned internal dilution in narrow ore. External unplanned HW and FW dilution was not applied within DSO due to the wide variation in resource thickness (0.2 m to 4.4 m) and dip (55° to 90°).

Deswik mine design software was used to refine these optimized blocks into mineable stope shapes and to apply the unplanned dilution and recovery referred to in Section 15.2. The stope shapes were evaluated on a complete mining-block basis to ensure that the mined material met the 5.0 g/t Au break-even cut-off grade parameter. Additionally, marginal stopes were evaluated for profitability based on unit-cost assumptions and operating development requirements. An ore-development incremental cut-off grade of 1.5 g/t Au was used, which accounts for the cost of processing and G&A.

15.6 Mineral Reserve Estimate

Table 15-3 presents the Mineral Reserve estimate for the Harte Gold Sugar Zone Mine as of 31 December 2020. Notes on the ore reserve are provided below the table.

Table 15-3: Mineral Reserve Estimate as of 31 December 2020

Classification	Zone	Tonnes ('000s)	Grade (g/t)	Ounces ('000s)
Probable	Sugar (North and South)	1,994	7.59	487
	Middle	1,460	6.62	311
	Total	**3,454**	**7.18**	**797**

Notes: The stated Mineral Reserves comply with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and are classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's CIM Definition Standards for Mineral Resources & Mineral Reserves. Mineral Reserve estimates reflect the Company's reasonable expectation that all necessary permits and approvals will be obtained and maintained.
Mineral Reserves are the economic portion of the Indicated Mineral Resources. Mineral Reserve estimates include mining dilution at grades assumed to be zero.
The 2020 Mineral Reserve estimate was prepared under the supervision of Mr. Chris McCann (P.Eng.), Director of Technical Services for the Company. Mr. McCann is a QP as defined by NI 43-101.
The Mineral Reserves were estimated as of 31 December 2020 using a gold price assumption of $1,450/oz.
A mining cut-off grade of 5 g/t has been estimated based on OPEX projections, sustaining capital development cost, mining dilution and recovery, royalty payment requirements, and applicable metallurgical recovery.
Totals may not add due to rounding.

16 MINING METHODS

16.1 Introduction

The Sugar Zone Mine mineralization typically comprises tabular, narrow veins, with economic resource widths varying from 0.2 to 4.4 m and dipping at around 70° to the southwest. Ground conditions are exceptionally good, allowing narrow longhole stopes to be mined with relatively low levels of unplanned dilution.

Access to the mine is via a surface portal and a single 4.3 m wide x 4.5 m high decline to about 100 m below the portal where the main decline divides into three separate FW declines that provide access to the North, South, and Middle Zones.

A standard fleet of mechanized equipment, comprising 2-boom and single-boom jumbos, 1.8 m^3 (2 yd) and 4.5 m^3 (6 yd) load-haul-dump (LHD) loaders, 30-tonne trucks, scissor decks, longhole drills, and various ancillary equipment, are employed in development and longhole open stoping.

In late 2020, the Operation transitioned from contractor to Owner mining for development and mining operations. Longhole drilling and blasting, and definition drilling continue as contractor operations.

The mine is forecast to produce at a rate of 800 t/d of ore in 2021 and to increase to 1,200 t/d in 2023. This increase will be driven primarily by full production rates being reached in the Middle Zone, which is being accessed and developed in 2021 and 2022. Paste backfill is planned for implementation in 2023 and will reduce the mobile equipment currently required for unconsolidated waste backfill.

16.2 Historical Mining

Mining operations commenced in 2016 with the accessing and trial mining of a 70,000-tonne bulk sample. This bulk sample was completed in March 2017 prior to completing the process plant. The bulk sample ore was trucked 80 km to Barrick's Williams Mine for processing.

Commercial production was achieved in January 2019. Figure 16-1 shows the as-built stoping and development as of 31 December 2020.

Table 16-1 shows the historical production for the operation. The pre-2018 production includes the bulk test stoping and the 2020 production is low due to a three-month COVID-19 shutdown.

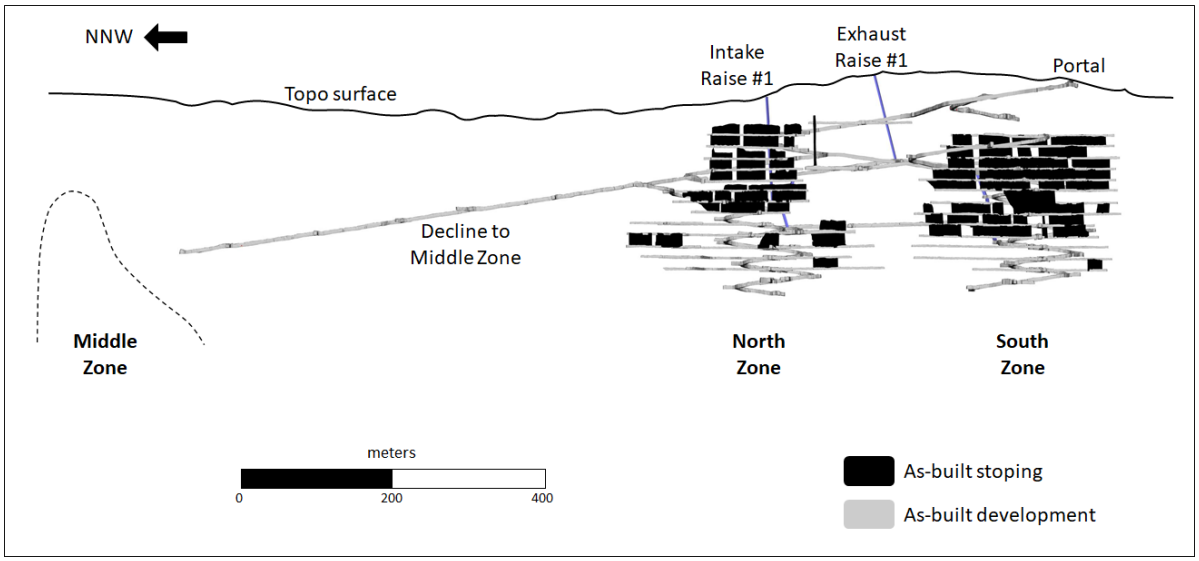


Figure 16-1: As-Built as of 31 December 2020

Table 16-1: Historical Production

	Waste Mined (t)	Ore Mined (t)	Grade Mined (g/t Au)	Ounces Mined (oz Au)	Waste Development (m)	Ore Development (m)
Pre-2018	96,000	97,853	5.3	16,674	1,600	1,625
2018	90,467	20,380	4.0	2,621	1,573	505
2019	191,737	179,385	5.2	29,123	3,161	3,046
2020	183,882	133,969	6.2	28,067	2,625	2,147

16.3 Geotechnical Design

In 2019, Mine Design Engineering Inc. (now RockEng Inc. [RockEng]) completed a feasibility study evaluation of the geotechnical conditions at the mine and provided stoping and development design recommendations (Mine Design Engineering Inc., 2019). This section summarizes the RockEng report and provides commentary on experience gained during mining in 2019 and 2020.

16.3.1 Stope Design

RockEng recommended sublevel intervals based on experience with the bulk sample stoping, which had a nominal level interval of 15 m. Further consideration was given to the average width of the mineralization and the need to minimize mining dilution. The average stope design width was approximately 1.8 m (1.5 m to 3.5 m range) for the North and South Zones, and 2.8 m (1.6 m to 6.7 m range) for the Middle Zone. Given the narrower average vein width of the Sugar Zone, 15 m sublevels were used to minimize unplanned dilution, while 20 m sublevel intervals were determined for the Middle Zone. Current experience indicates that the minimum achievable fully diluted stope width is

2.0 m in the North and South Zones, and all stopes in the current design have been diluted to a minimum 2.0 m wide.

RockEng recommended stope strike dimensions at 40 m in the North and South Zones and 25 m in the Middle Zone to generate hydraulic radii (HR) of 6 to 6.2. Original designs include HW cable bolting fanned from the sill drives, which reduces the unsupported HW HR to approximately 5 m and plots within the Stable Zone of the modified stability graph for the majority of expected ground conditions. Mining during 2019 and 2020 did not incorporate any HW cable bolting, and there is no indication of significant HW structural failure that would be ameliorated by cables. HW cable bolting will likely not be required until mining depth exceeds 500 m, and then only in areas of poor ground conditions.

Numerical modeling by RockEng has shown that based on the current mining depths and relatively high unconfined compressive strength of the host rocks there is little risk of heavy, stress-induced rock mass damage and associated seismicity. The low level of risk associated with stress-induced damage, and associated seismicity for the Sugar Zone Deposit will allow operational sequencing flexibility.

Based on geotechnical mapping and core logging by RockEng, the typical rock mass quality is expected to be good (Q' = 11 to 24).

Empirical stope stability analyses by RockEng concluded that, for typical conditions, stopes are expected to perform well:

- Stress in the major influence on stability variability:
  - Above 200 m mining depth stress conditions are favourable (A factor is 0.8–1.0).
  - Between 200 and 500 m mining depth is a transition zone.
  - Below 500 m mining depth, stress conditions are less favourable (A factor drops to 0.4–0.1).
- HW geometry becomes more favourable with depth as the dip steepens (C factor increases from 5.7–7.9).
- Side walls and backs are expected to perform well due to low HR.
- HW stability will be the control factor for design.
- Current stope dimensions are expected to see good HW performance, and routine cable bolting of the HW is not expected to be required.
- There is an opportunity for larger stopes with cable bolting.

In general, stope dimensions are conservative to minimize the potential for overbreak dilution. It is anticipated that there will be opportunities to optimize the stope dimensions as further operational data are collected, and cable bolts can be used selectively, based on individual stope conditions. The bulk sample trial stopes, while relatively shallow, provided evidence of large, stable stoping voids—approximately 40 x 40 m HW exposures (HR of approximately 10 m)—with minimal sloughing over an extended time.

Figure 16-2 shows an HW stability assessment for unsupported Sugar Zone and Middle Zone stopes based on stope dimensions in the mine plan.

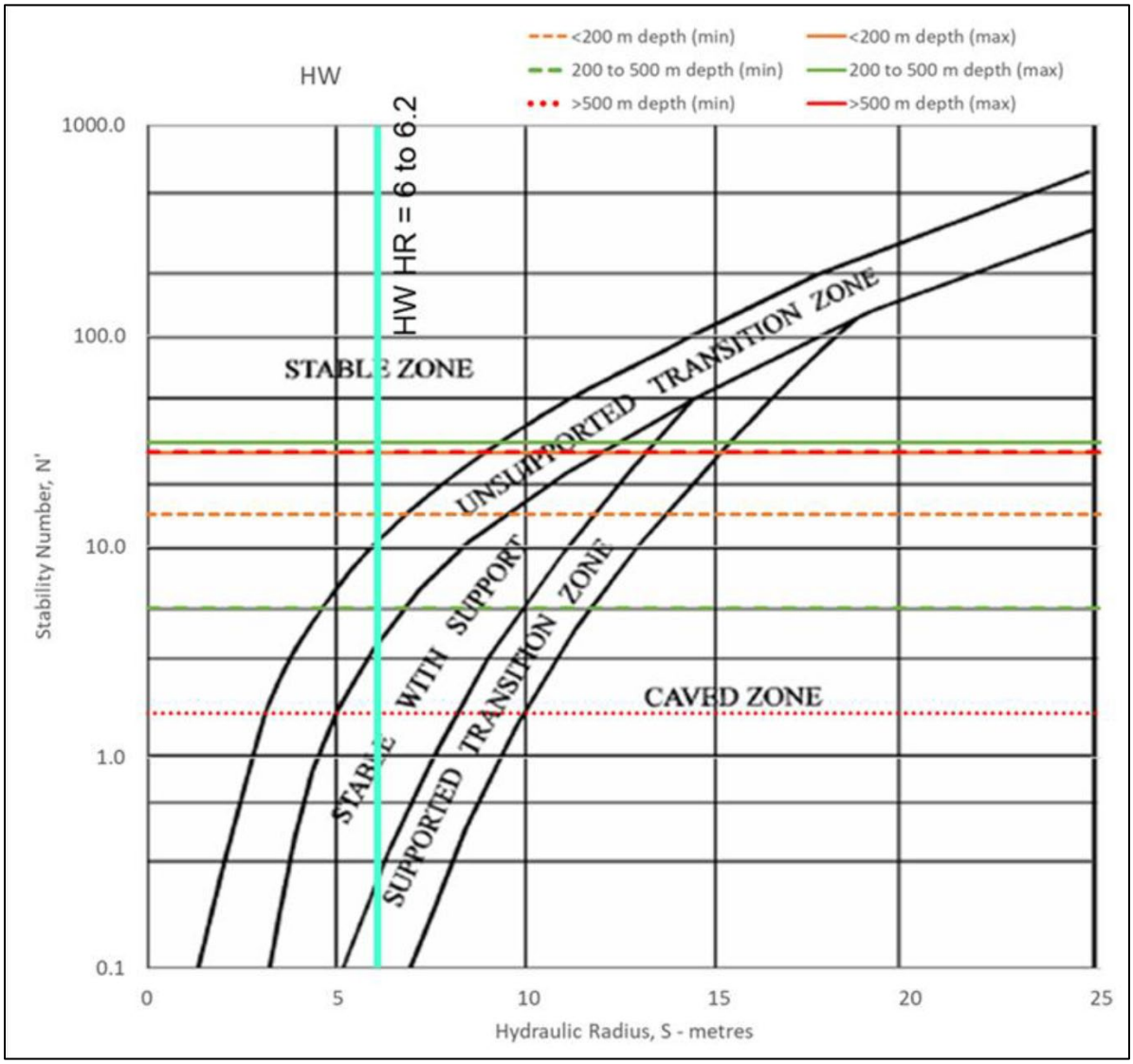


Figure 16-2: Empirical Assessment of Stope Hanging Wall

RockEng analysis (incorporating the mine design, rock strength data, geomechanical site characterization, and numerical stress modelling) determined the probability of occurrence for overbreak for each empirical bin of equivalent linear overbreak and sloughage (ELOS), and three depth ranges (0 to 200 m, 200 to 500 m, and greater than 500 m). The results for supported HWs are provided in Table 16-2. As expected, ELOS increases with depth; however, in the majority of stopes the supported HWs are expected to perform well, and display blast damage levels of ELOS only. While this study assumed constant stope sizes for North, South, and Middle Zones, actual mine operations will be able to optimize individual stope dimensions for conditions, to minimize dilution and maximize

stoping efficiency. A minimum average FW-plus-hanging-wall combined ELOS of 0.5 m was applied to all stopes.

Table 16-2: Probability of Hanging Wall Failure for Supported Conditions

ELOS (m)	North and South Zones			Middle Zone		
	Depth <200 m	Depth 200 m–500 m	Depth <500 m	Depth <200 m	Depth 200 m–500 m	Depth <500 m
<0.5	98	92	92	93	71	71
0.5–1	2	6	6	6	20	20
1–2	0	2	2	1	7	7
>2	0	0	0	1	2	2

16.3.2 Development Support Design

Harte Gold developed and optimized the ground support standards based on RockEng's empirical and kinematic analyses.

The spans for single headings are measured as the horizontal span. Spans in intersections are measured as the diameter of the largest circle that can be drawn inside the plan of an intersection, allowing for a total of 0.5 m of overbreak at corners.

The ground support standard for waste development is as follows:

- M20 resin-grouted rebar installed in a 3-2-3 pattern, 8 bolts per sheet of screen (equivalent to a 1.2 x 0.75 m staggered spacing) with appropriate steel plates
  - Span 4.3 m—1.5 m bolt length (normal—ramp advance)
  - Span 5.0 m—1.8 m bolt length
  - Span 7.0 m—2.4 m bolt length (intersections)
  - Span 8.0 m—3.1 m bolt length (intersections)
  - Span >8.0 m—To be specified by engineer on development layout.
- Mesh size is 1.5 x 3.4 m.
- Support must be installed across the back and down walls to 1 m below the base of the shoulder.

The ground support standard for sill development is as follows:

- M20 resin-grouted rebar installed in a 3-2-3 pattern, 8 bolts per sheet of screen (equivalent to a 1.2 x 0.75 m staggered spacing) with appropriate steel plates
  - Span 3.6 m—1.2 m bolt length
  - Span 4.6 m—1.5 m bolt length
  - Span 5.5 m—1.8 m bolt length
  - Span 6.5 m—2.4 m bolt length
  - Span 7.3 m—3.1 m bolt length
  - Span >7.3 m—To be specified by engineer on development layout.
- Mesh size is 1.5 x 3.4 m.
- Support must be installed across the back and down walls to 1 m below the base of the shoulder.

16.4 Backfill Design

In mid-2020, P&C retained RockEng to provide a feasibility-level geomechanical design for the paste backfill strength recommendations. RockEng has evaluated the following cemented paste backfill strength requirements, and Section 16.16.1 presents the results of this evaluation.

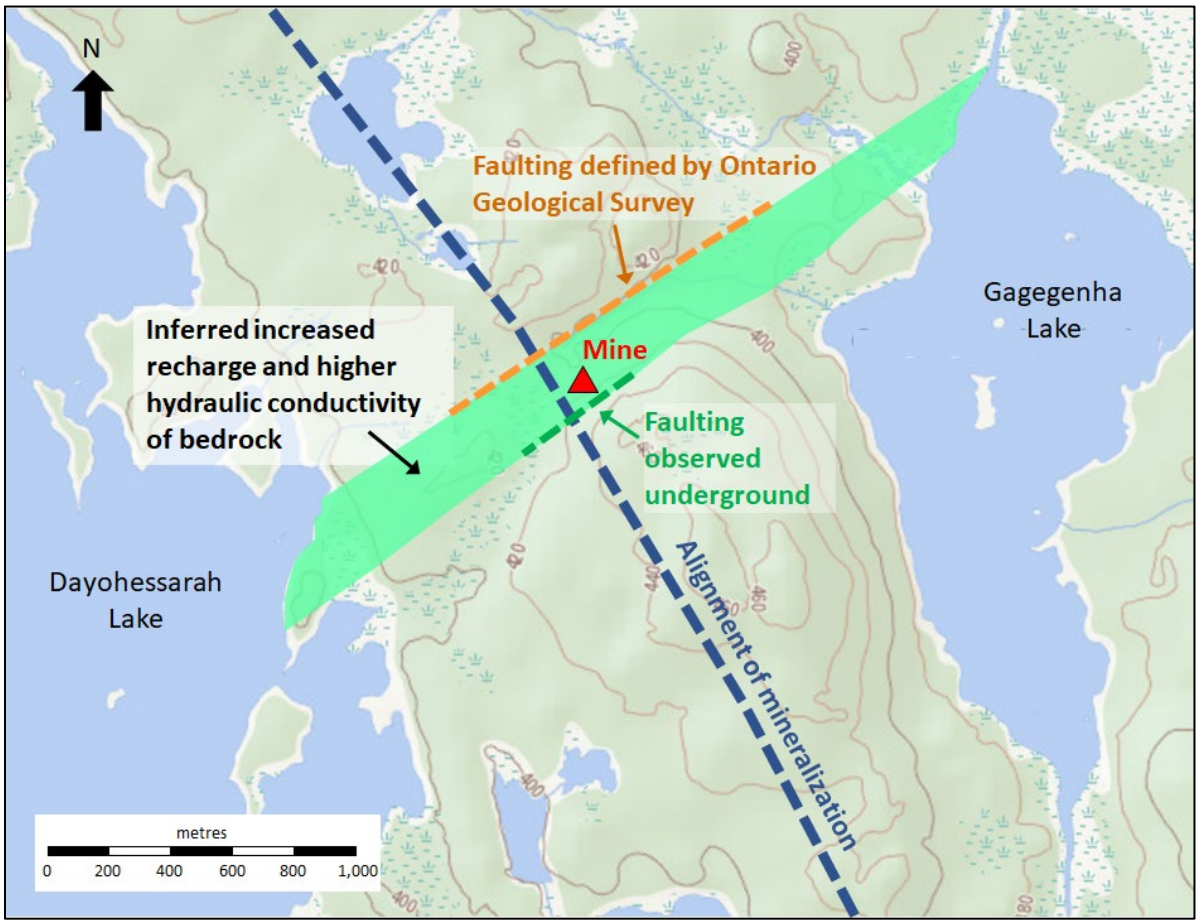
16.5 Hydrogeological Considerations

Subsurface soil conditions at the mine comprise competent bedrock, overlain by a discontinuous, thin veneer of silty sand that ranges from 0 to 5 m thick. Based on the observed groundwater elevations at the Project site, groundwater flow directions generally follow surface topography. Based on hydraulic conductivity values, the upper, weathered bedrock is connected to the overlying overburden with values ranging from 10^{-5} to 10^{-6} m/s. Bedrock beyond 5 m deep is less permeable, with hydraulic conductivity values ranging from 10^{-8} to 10^{-9} m/s.

Amec Foster Wheeler (now Wood Canada Limited [Wood]) developed a numerical 3-D steady-state groundwater flow model to predict groundwater inflows to the mine workings. The model uses the physical bedrock and overburden aquifer parameters collected from baseline hydrogeological characterization studies, packer testing on diamond drill holes, and interpretation of fault zones. The most recent modelling was undertaken in 2018 using MODFLOW (Version 4.6), developed by Schlumberger Water Services; considerations in the updated model are listed below:

- Given the advanced stage of the underground workings, bedrock hydraulic conductivity was determined using pumping records for discrete time spans, with accompanying drawings of the mine workings to evaluate void spaces during the selected times.
- Fracture zones encountered in the underground workings projected to surface, and the faults mapped by the Ontario Geological Survey (Stott, 1999) are shown in Figure 16-3. To account

for these features in the model, a zone of increased hydraulic conductivity was introduced in the area between these features (approximately 200 m wide). In addition to introducing the higher hydraulic-conductivity zone that coincides with the observed faulting near the Project Site, a zone of increased recharge was introduced to account for the potential increase in recharge that may occur in this zone.



Source: Modified from Stott, 1999.

Figure 16-3: Zone of Modified Hydraulic Conductivity and Increased Recharge Rate

Model results indicate the long-term inflows into the fully developed underground workings will be in the order of 410 to 470 m^3/d once steady state is achieved. Additionally, no impacts to surface water features are anticipated, and the remote location precludes interference with any groundwater users. Based on daily inflows measured in 2020, the forecast average inflow for 2021 is 380 m^3/d, with strong seasonal variability. Average daily inflows for 2020 are summarized in Table 16-3.

Table 16-3: Average Daily Groundwater Inflows to Mine Workings

Year	2020											
Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average Daily Dewatering (m^3/d)	170	225	250	550	750	610	380	210	290	440	290	170

Surface diamond drill holes represent a risk of increased hydraulic connectivity with overburden aquifers and surface water features; these are identified on mine plans and grouted as part of standard operating procedure.

Water table drawdown is measured at established observation wells on an on-going basis and the groundwater flow model is calibrated on a regular basis to determine if the usage has an increased potential to impact surface water features, or if the allowable withdrawal rate needs to be adjusted in the Permit to Take Water that has been issued to authorize mine dewatering.

16.6 Mining Method

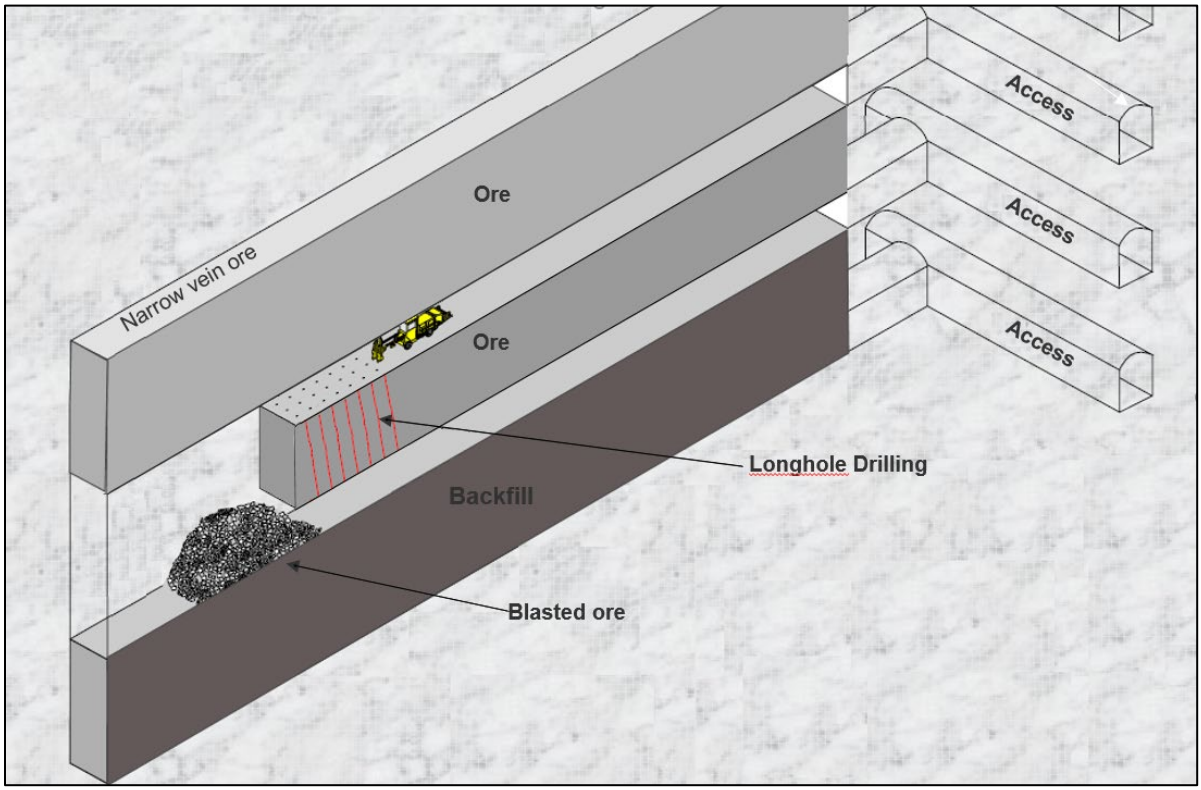
The North, South, and Middle Zone mineralization typically comprises tabular, narrow veins, approximately 1.5 to 3 m thick, dipping at around 70° to the southwest. Excessive dilution is potentially a major risk to operating success in narrow mineralization. Longitudinal longhole retreat stoping was selected as appropriate for this deposit based on the favourable geometry, current geotechnical knowledge, and success of ongoing operations. Both the ore and the host rock are sufficiently competent to facilitate the void sizes required for effective longhole stoping. A schematic of the longitudinal longhole retreat mining method is provided in Figure 16-4.

The longhole open-stoping mining method provides the following advantages:

- Safe—personnel do not enter the production mining voids (remote-controlled equipment only).
- Proven—many instances of the methods application throughout the world.
- Highly mechanized—reduces personnel exposure to health and safety risks.
- Easy access to resources—skilled labour and equipment are readily accessible locally and around the world.
- Low cost—leverages off large drill and blast scale and minimization of development.

Paste backfill will be the primary fill type after paste plant construction in 2023, and offers advantages, by:

- Reducing the need for rock pillars, maximizing ore recovery
- Reducing the duration of the stoping cycle
- Reducing the number of diesel-powered loading and hauling units underground, thereby reducing the overall primary ventilation requirements
- Reducing the size of the surface tailings storage facility.

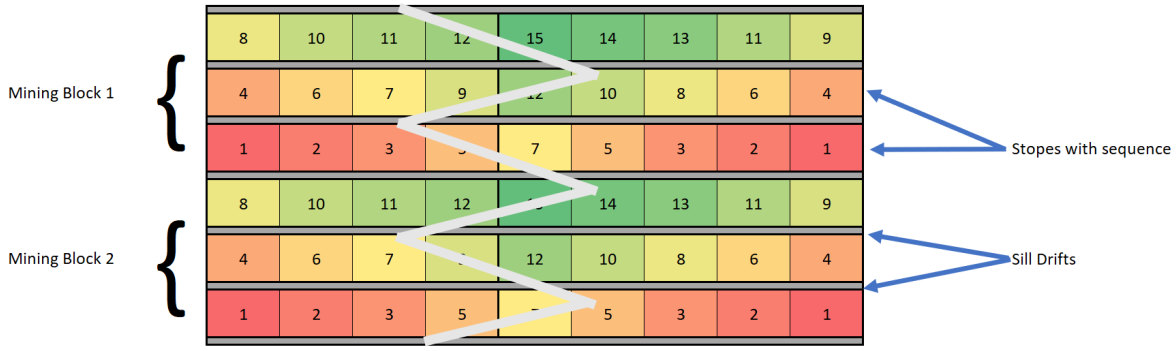


Source: Modified from AMC Mining Consultants by Harte Gold

Figure 16-4: Longitudinal Longhole Retreat Mining Method

Typically, three sublevels will be combined to form a mining block (Figure 16-5). Within each mining block a bottom-up stoping sequence will be used, with mining commencing on the lowest sublevel of the block and progressing inward and upward in a chevron pattern. Blind uphole-retreat longhole stoping will be used to extract the uppermost level of each mining block (which will usually be directly underneath a previously mined-out and filled mining block).

Two options are available for mining under the previously mined block. The first is to leave a 2 m sill pillar in place to prevent paste backfill from failing and diluting the stope below. The second option is to increase binder content in the paste backfill in the level immediately above to ensure full recovery of ore using blind uphole retreat without the need for a rock sill pillar. This Technical Report uses the first option, and all the uppers stopes have an 85% recovery applied, to simulate leaving a 2 m sill pillar in place. There is potential to increase ore recovery through implementing the second method, but this decision will be made once the paste backfill system is operating.



Source: Harte Gold

Figure 16-5: Mining Sequence Schematic

Subdividing the mine into mining blocks increases the number of mining faces available, thereby increasing reliability and flexibility of production. It will also maximize the production rate.

Continued ramp pre-production development will provide access to new mining blocks as upper mining blocks are depleted. The rate of ramp pre-production capital development will be critical in achieving and sustaining the production rate.

The paste backfill plant is designed to generate paste backfill at a nominal design flow rate of 32 m^3/h or up to 700 m^3/d at 92% mechanical availability and 100% utilization of availability. The schedule calls for an average total paste backfill rate of 240 m^3/d beginning in 2023. The difference between the requirement of 240 m^3/d and the design capacity of 700 m^3/d is to account for the fact that planned and unplanned downtime for mechanical repairs and maintenance is reasonably expected in the plant, and the underground mine will not always have stopes available to receive pastefill. A positive-displacement pump and piping system will transfer paste from the plant to the twinned vertical borehole system for underground distribution. Paste backfill barricades will be constructed in the lower sill drives with waste rock packed tightly into the stope entrance using a rammer attached to an LHD. Shotcrete may be used on the brow of the barricade to provide a tight seal, and in some instances engineered cable bolt, mesh, and shotcrete barricades may be used where space is limited.

16.7 Underground Development Design

The mine layout is based on ramp development from surface to access the mine workings and production levels. Ramp development will extend down from both the existing North and South Zones. Two ramps are required in this area due to the large strike extent (up to 650 m), while a single ramp access is planned for mining the Middle Zone. The two Sugar Zone ramps incorporate several linking ramps at approximately 180 m vertical intervals that will facilitate ventilation, pumping, equipment movement, and escape routes.

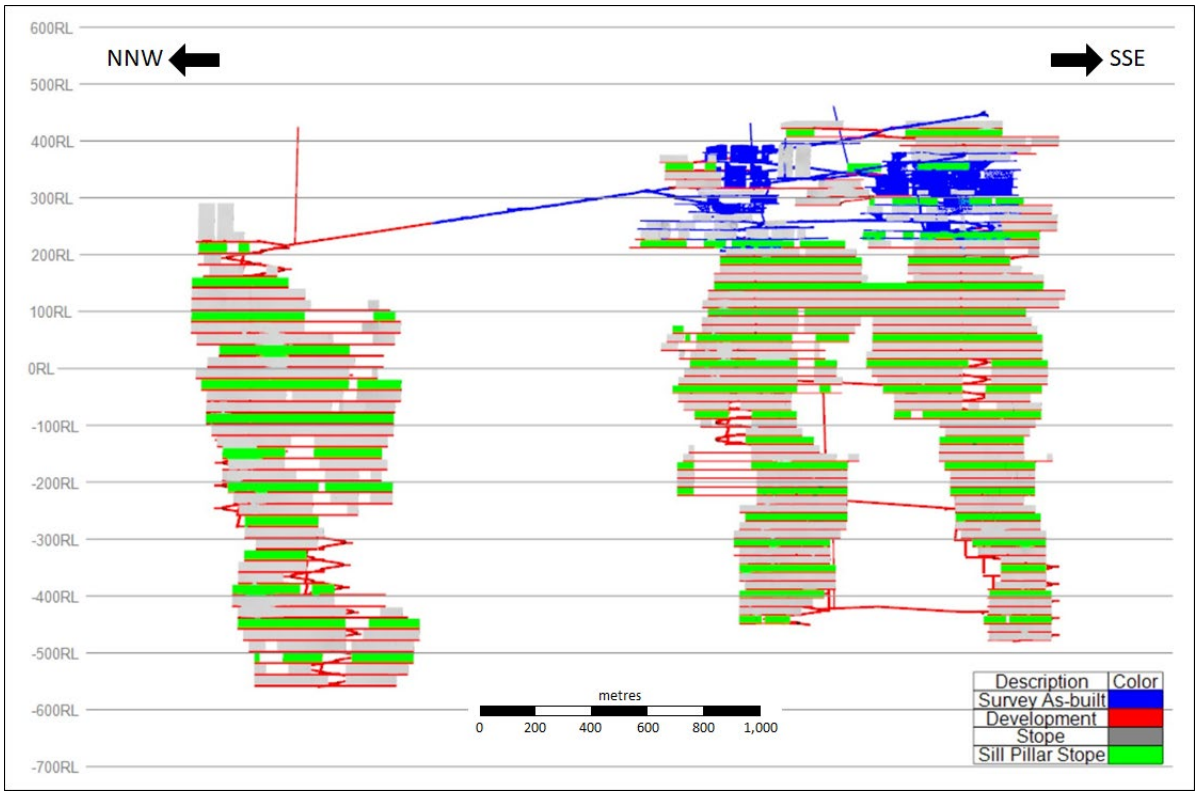


Figure 16-6: Mine Layout Longitudinal Projection

The twin ramp layout for Sugar Zone has the following advantages:

- Significantly reduces the primary airflow velocities (reduced power costs and fan CAPEX costs)
- Reduces congestion from traffic
- Provides additional escapeway (secondary egress) routes
- Offers increased drill coverage for definition and exploration drilling
- Increases the number of available mining blocks (independent producing areas), resulting in greater production reliability
- Increases production sequencing options for improved geotechnical sequencing
- Decreases the ore haulage distances (improved loading productivity)
- Decreases waste development required along strike
- Eliminates FW drifts where they might otherwise have been required.

Table 16-4 presents the ramp design criteria.

Table 16-4: Ramp Development Criteria

Item	Criteria
Ramp Gradient	15% (with moderate reduction in gradient adjacent to access)
Ramp Radius of Curvature	20 m (centreline)
Ramp Re-Muck Bay Interval	120 m nominal

Mine level development typically comprises the following:

- Level access drifts—developed from the ramp to the zone
- Ore (sill) drifts—developed in mineralization along the strike of the zone
- Re-muck bays—at regular intervals along the ramp and on the access drifts
- Ventilation drifts
- Ventilation raises
- Sump excavations.

Level access drifts will be developed from the ramps to intersect the ore zone, and subsequently branch into sill drifts that will be developed north and south along strike. Level accesses will include sumps for drainage and stockpiles for ore storage and truck loading. Ventilation raises will be developed between levels, either from the access or ramp, to extend the primary vent circuit as the mine is developed to depth.

Sumps will be located along the level access development on all sublevels. All sublevel development will be at a positive gradient to facilitate drainage to the sublevel sump. Sublevel sumps will be aligned vertically between levels so that sequential drainage can occur down through each sump to permanent or temporary pumping stations located at regular intervals throughout the mine.

The level intervals were selected primarily to mitigate dilution from the longhole open-stoping method. Typically, increasing level spacing provides cost and productivity benefits at the expense of drilling accuracy, void dimensions, void stability, and hence dilution. Given the average ore thicknesses of the Sugar Zone and Middle Zone at 1.1 and 2.2 m, respectively, the Sugar Zone is more susceptible to dilution than the Middle Zone. As such, level intervals of 15 m in the Sugar Zone and 20 m in the Middle Zone were selected as a balance between dilution, productivity, and cost.

A typical sublevel layout for the Sugar Zone is provided inFigure 16-7. The 15 m sublevel spacing and ramp minimum design radius of 20 m has resulted in a spiral ramp layout that requires alternate level accesses to be approximately 40 m longer than the other.

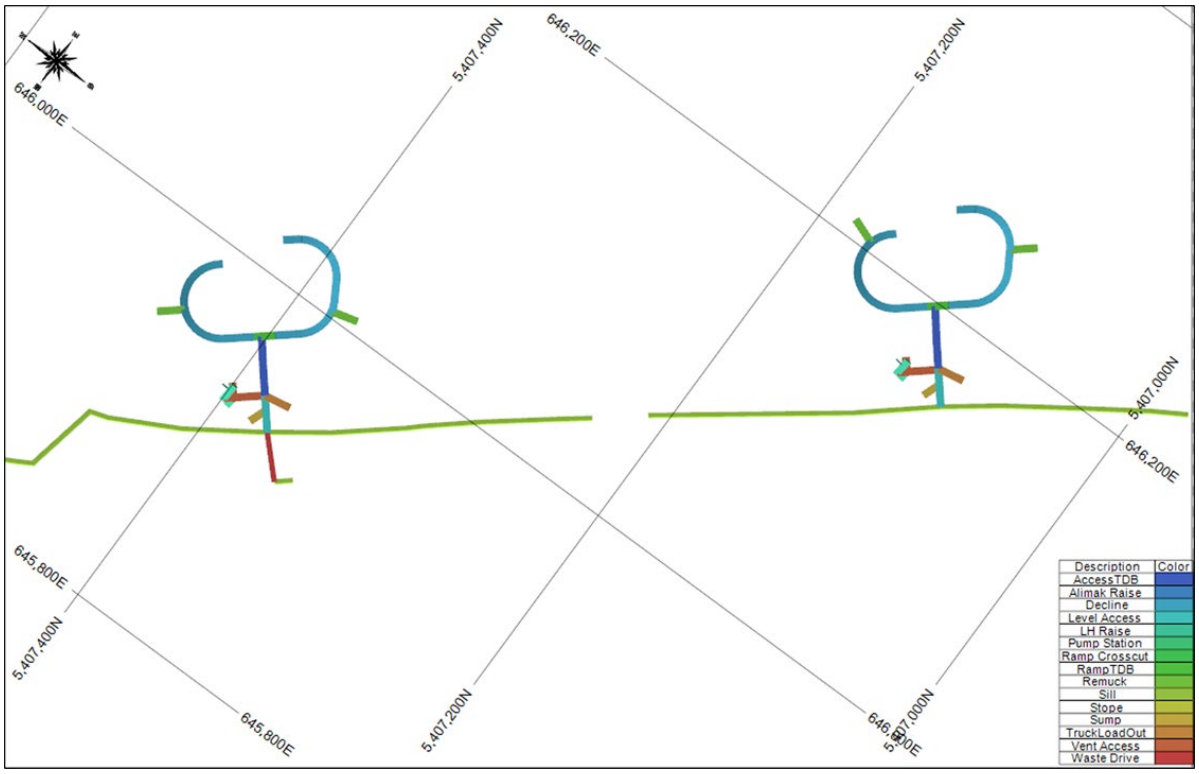


Figure 16-7: Sugar Zone Typical Level Plan

A typical sublevel layout for the Middle Zone is provided in Figure 16-8. The 20 m sublevel spacing, and ramp minimum design radius of 20 m have resulted in a spiral-loop ramp layout that provides shorter sublevel accesses on average than the Sugar Zone sublevel accesses.

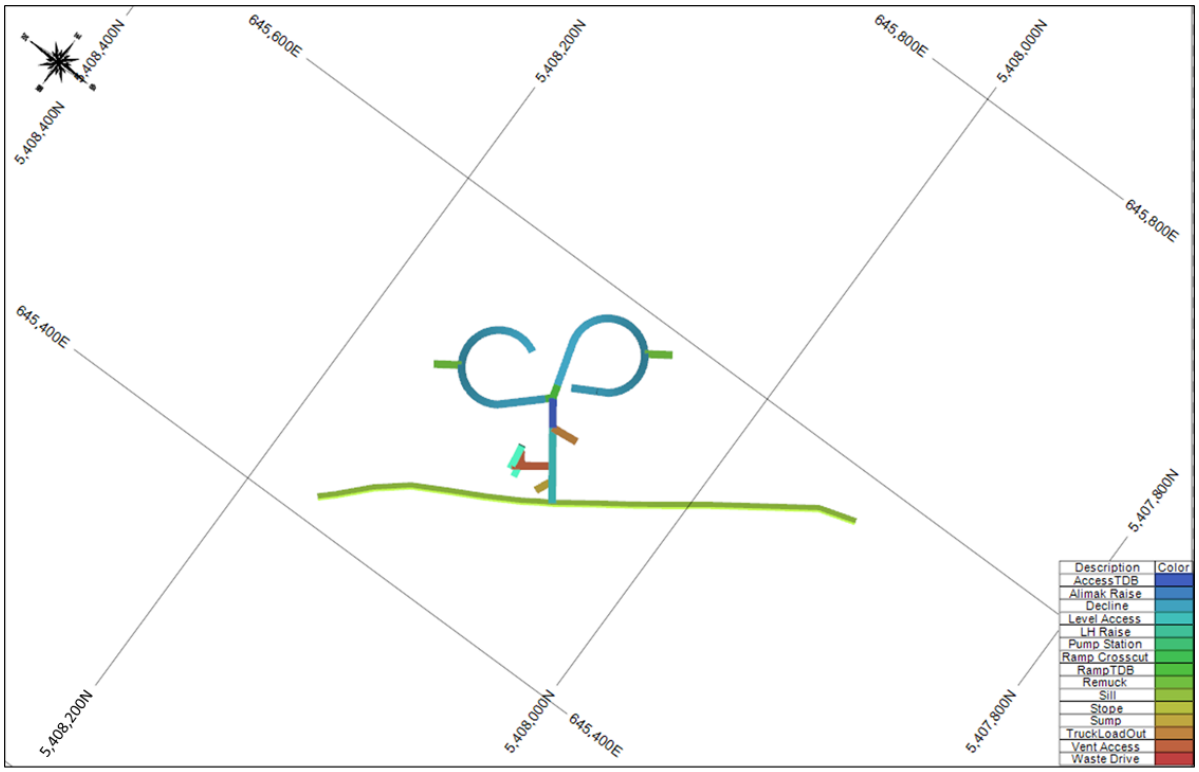


Figure 16-8: Middle Zone Typical Level Layout

The lateral and vertical development dimensions are provided in Table 16-5 and Table 16-6, respectively. Alimak raises will be developed upwards using a first-pass nominal cross-section of 2.4 x 2.4 m and slashed to 3.7 x 3.7 m on a second pass. Note that larger Alimak raise dimensions will be employed if possible, to further reduce the resistance of the primary air circuit.

Table 16-5: Lateral Development Profiles and Gradients

Development	Dimensions	Profile	Gradients
Ramp	4.3 m wide, 4.5 m high	Rectangular	15%
Re-Muck	4.3 m wide, 4.5 m high, 15 m deep	Rectangular	5%
Re-Muck Intersection Stripping	4.3 m wide, 6.5 m high	Rectangular	5%
Sublevel Access	4.3 m wide, 4.5 m high	Rectangular	2% to 5%
Ventilation Access	4.3 m wide, 4.5 m high	Rectangular	2% to 15%
Truck Load Out Stations	4.3 m wide, 6.5 m high, 15 m deep	Rectangular	2% to 5%
Ore Drift (Sill Drift) NZ and SZ	3.0 m wide, 3.0 m high	To HW contact	1%
Ore Drift (Sill Drift) MZ	4.0 m wide, 4.0 m high	To HW contact	1%
Refuge Chamber Cut-Out	4.3 m wide, 4.5 m high, 15 m deep	Rectangular	2%
Safety Bays	2 m wide, 2 m high, 2 m deep	Rectangular	0%
Sump	4.3 m wide, 4.5 m high, 10 m deep	Rectangular	20%
Transformer Station	5.5 m wide, 4.5 m high, 9.5 m deep	Rectangular	2%

Note: The mine design uses flat gradients for all level accesses and sill development to simplify the design exercise.

Table 16-6: Vertical Development Profiles and Gradients

Development	Dimensions	Profile	Gradients
Ventilation Raise—Alimak	3.7 m x 3.7 m	Square	60°–90°
Ventilation Raise—Longhole	4.0 m x 4.0	Square	60°–90°

16.8 Stope Design

The stoping cut-off grade estimation, optimization, and design process is discussed in Section 15.

16.9 Longhole Drilling and Blasting

Drilling and blasting are carried out by a contractor who supplies supervisory and operational personnel and air-powered longhole drills. The following points provide a general view of the parameters used in drilling and blasting the longhole stopes:

- Drill-hole diameter of 64 mm (2.5).
- Ring burden of 1.1 m.
- Downhole rings are aligned vertically.
- Uphole rings are dumped forward at angle of ±70° from horizontal.
- Holes within rings are drilled parallel where possible:

- Holes are drilled parallel to the HW and FW contacts.
- The hole closest to the HW in each ring is drilled 0.3 m inward from the interpreted HW contact where the vein is wide enough to allow it; where the vein is narrower than the minimum drilling pattern, the hole closest to the HW is drilled on or outside of the interpreted HW contact.
- The hole closest to the FW in each ring is drilled on or slightly inside the interpreted FW contact where the vein is wide enough to allow it; where the vein is narrower than the minimum drilling pattern, the hole closest to the FW is drilled outside the interpreted FW contact.

- A "Dice 5" drill-hole pattern is used where the stope true width is less than 2.0 m. A "Dice 5" pattern comprises two holes per ring, with a single additional hole drilled between rings.

Typical longhole drill designs for ore widths less than 2.0 m and greater than 2.0 m are provided in Table 16-7.

Table 16-7: Drill and Blast Design Criteria—Production Rings

Item	Unit	Width <2.0 m	Width >2.0 m
Bulk Explosive	type	ANFO / Emulsion	ANFO / Emulsion
Bulk Explosive Density	t/m³	0.8 ANFO / 1.15 packaged	0.8 ANFO / 1.15 packaged
Drill Hole Diameter	mm	64	64
Ring Dump Angle	degrees from horizontal	90 downholes	90 downholes
		70–80 upholes	70–80 upholes
Detonators per Hole	no.	2	2
Boosters per Hole	no.	2 x 200 g	2 x 200 g
		(or 340 g)	(or 340 g)
Holes per Ring	no.	2.5 (2 with 1 hole between rings)	3
Ring Burden (Horizontal Distance)	m	1.1	1.1
Volume Blasted per Drill Metre	m³/m	0.6	0.9
Ore Bulk Density	t/m³	2.7	2.7
Tonnes Blasted per Drill Metre	t/m	1.8	2.4
Percentage of Hole Charged	%	90	90
Explosive Mass per Hole Metre	kg/m	2.3	2.3
Powder Factor	kg/t	1.3	1.0
Re-Drills	%	3.3	3.3
Drilling Factor Including Re-Drills	t/m drilled	1.7	2.3

16.10 Material Movement

Small, diesel, remote-controlled LHD units (1.8–2.5 m^3 capacity) load the blasted ore from ore development and stoping sources to re-muck bays located on the level access. Larger, diesel LHD units (4.5 m^3 capacity) subsequently re-handle ore from the re-muck bays into 30-tonne diesel trucks for hauling to the surface run-of-mine (ROM) pad. Waste rock is handled similarly, without the need for smaller LHDs, given the larger dimensions of the waste headings.

16.11 Mine Mobile Equipment

Table 16-8 presents the primary and support mobile mine equipment for the underground mine. Two-boom jumbos are used to drill waste development rounds. Single-boom jumbos are used to develop the ore sills. Ground support is installed using stopers on scissor decks in waste development and from the floor in ore sills. Mucking ore from sills and stopes to remucks requires 1.8 m^3 LHDs; 4.5 m^3 LHDs load 30-tonne trucks from remucks and from waste development faces.

Table 16-8: Equipment List

	2021	2022	2023	2024	2025	2026	2027	2028	2029
Equipment List	42	42	52	53	54	54	54	54	42
Two-Boom Jumbo	3	3	3	3	3	3	3	3	2
Single-Boom Jumbo	2	2	2	2	2	2	2	2	2
Scissor Deck	4	4	4	4	4	4	4	4	2
30-tonne Trucks	5	5	6	7	8	8	8	8	7
4.5 m^3 LHD	4	4	5	5	5	5	5	5	4
1.8 m^3 LHD	6	6	8	8	8	8	8	8	6
Primary. Equip.	24	24	28	29	30	30	30	30	23
Explosives Loader	1	1	2	2	2	2	2	2	2
Minecat Fuel/Lube	2	2	2	2	2	2	2	2	1
Minecat w/Backhoe	2	2	3	3	3	3	3	3	2
Minecat w/Loader	1	1	2	2	2	2	2	2	1
Minecat w/Mast	1	1	1	1	1	1	1	1	1
Minecat w/X-Lift	1	1	1	1	1	1	1	1	1
Boom Truck	1	1	1	1	1	1	1	1	1
Grader	1	1	1	1	1	1	1	1	1
Telehandler	1	1	2	2	2	2	2	2	1
Toyota	1	1	2	2	2	2	2	2	2
Kubota	6	6	7	7	7	7	7	7	6
Support Equipment	18	18	24	24	24	24	24	24	19

16.12 Mine Personnel

Section 21.2.2.1 presents a summary of the mine personnel requirements over the LOM.

16.13 Productivity Estimates

The task rates used for scheduling are provided in Table 16-9. These rates are applicable to each individual activity in the schedule and are not intended as mine average rates, which will be lower (considering resource availability and dependencies between activities).

Table 16-9: Mining Activity Task Rates

Task	Productivity	Comment
Development—Ramps	2.25 m/d	Ramps only
Development—Ore Drifts	2.25 m/d	Ore drifts only
Development—Lateral	2.25 m/d	All other lateral development
Alimak Raise	3.6 m/d	Includes pilot and stripping phases
Longhole Drop Raise 4 x 4 m	3 m/d	
Stope Production Loading—SZ	250 t/d	
Stope Production Loading—MZ	250 t/d	
Longhole Drilling 64 mm Diameter	130 m/d	
Longhole Slot Drilling 64 mm Diameter	3 d	
Paste Backfilling	600 t/d (wet)	Limited to one stope at any time

Schedule optimization was completed using Schedule Optimization Tool (SOT) to produce a schedule with improved net present value (NPV) through iterations of a genetic algorithm.

To constrain the schedule optimization, several periodic production rate constraints were introduced, including:

- 750 m/month of lateral development
- 290,000 t/a of total mined material per each ramp (North Zone, South Zone, Middle Zone)
- Mill constraints of:
  - 25,000 t/month through Q3 2022
  - 28,000 to 34,000 t/month ramping up through Q4 2022
  - 36,000 t/month beginning Q1 2023.

The SOT schedule results and quantities were adjusted to align with the 2021 Budget mine plan.

16.14 Production Schedule

A chevron bottom-up stope sequence was applied to the stopes within each mining block. The chevron approach has the advantage of permitting increased mining flexibility by completing concurrent activities on multiple horizons. The stope sequence within a mining block is summarized in Figure 16-9. For each stope a sub-sequence of schedule activities exists as follows: slot/drop raise preparation, longhole drilling, loading, blasting, mining, and backfilling. An example of a stope activity subsequence from the schedule is provided in Figure 16-10, which also shows several examples of a

delay occurring between the completion of loading (Stope) and paste backfilling (PASTEFILL). This is a direct result of equipment-allocation levelling limiting paste backfilling to only one PASTEFILL activity at any point in time.

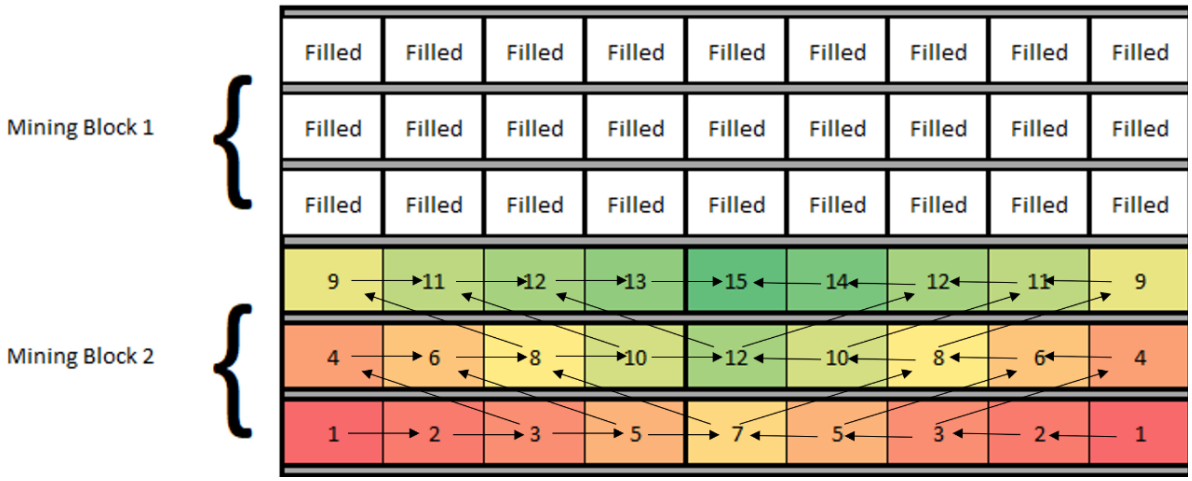


Figure 16-9: Stope Sequence within Mining Block

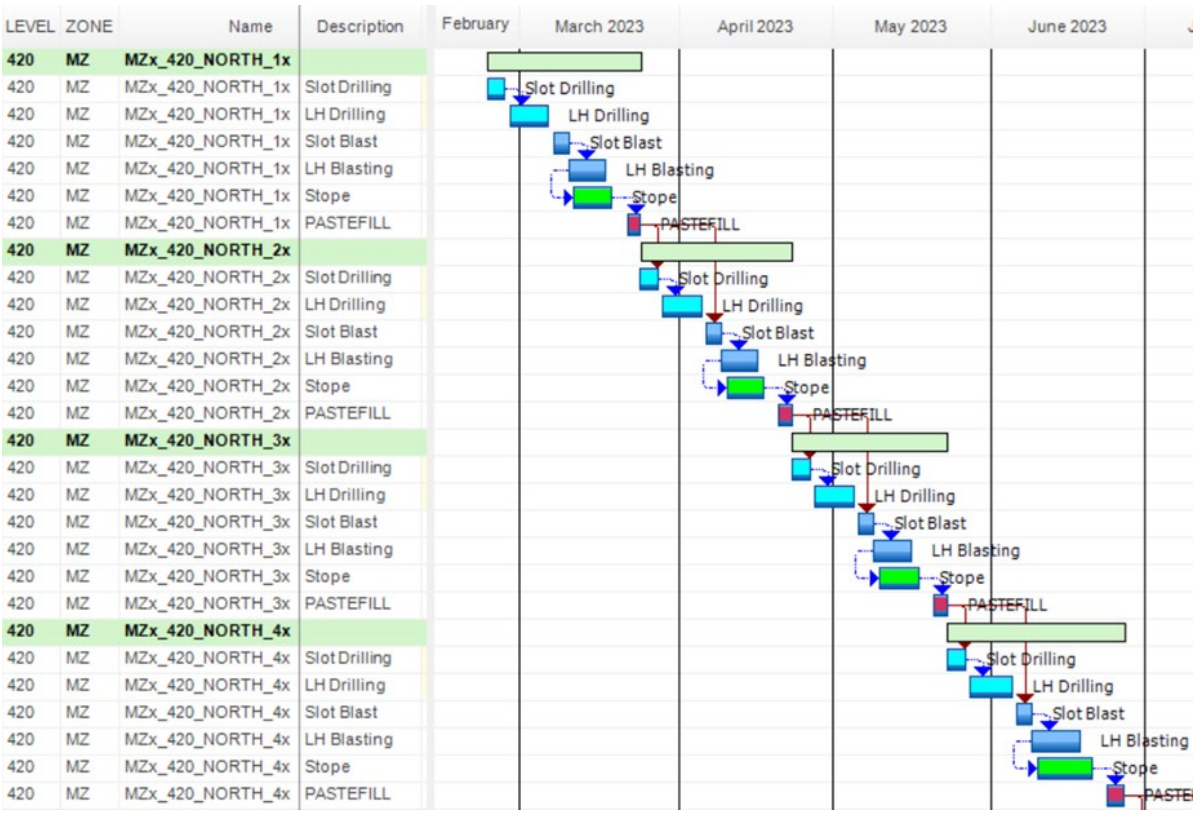
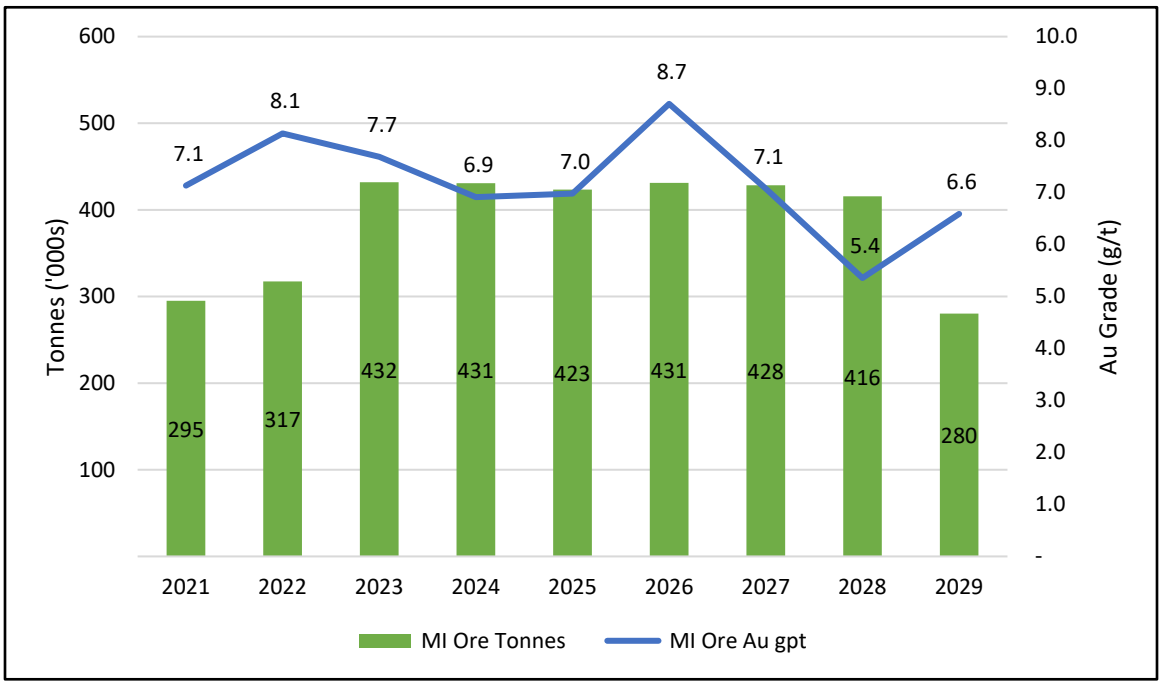


Figure 16-10: Gantt Chart Schedule with Stope Activity Sequence

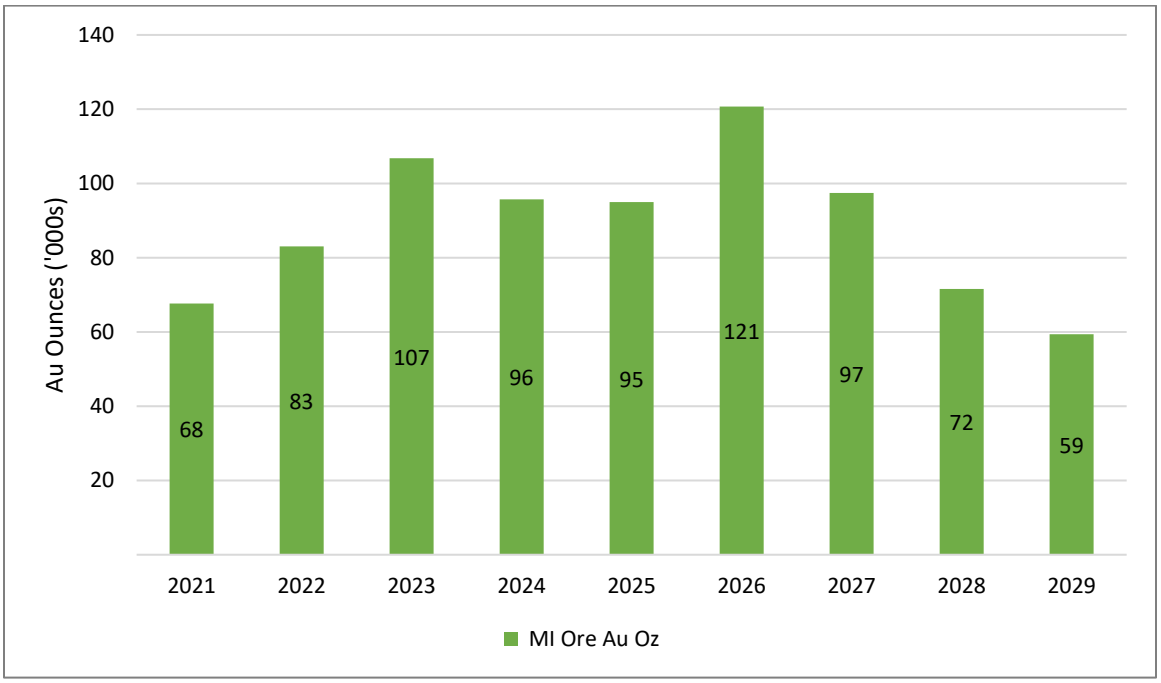
The mine production profiles for ore tonnes and grade, and contained gold are provided in Table 16-11 and Table 16-12, respectively. The following can be observed:

- Production rate of 1,200 t/d is sustainable after 2023, with a gradual ramp-down beginning Q4 2028.
- Grade profile is impacted by availability of high-grade zones (spatially driven).
- Low-grade areas are lower priority and are generally delayed to later in mine life.
- Contained gold production averaging 103,000 oz between 2023 and 2027 at a mining rate of 1,200 t/d, with an additional four years at lower gold production (2021 & 2022 at 800 t/d and 2028–2029 mining low grade and production ramp down).
- Annual peak of 121,000 oz of contained gold mined in 2026.



Note: The "MI" in the series names above stands for "Measured & Indicated" and is the prefix given to the interrogation fields used in Mineral Reserve generation.

Figure 16-11: Mined Ore Production and Gold Grade Profile by Year



Note: The "MI" in the series names above stands for "Measured & Indicated" and is the prefix given to the interrogation fields used in Mineral Reserve generation.

Figure 16-12: Mined Contained Gold Ounces by Year

Lateral development is relatively consistent over time, and composed of approximately 48% ore development and 52% waste development, as shown in Figure 16-13.

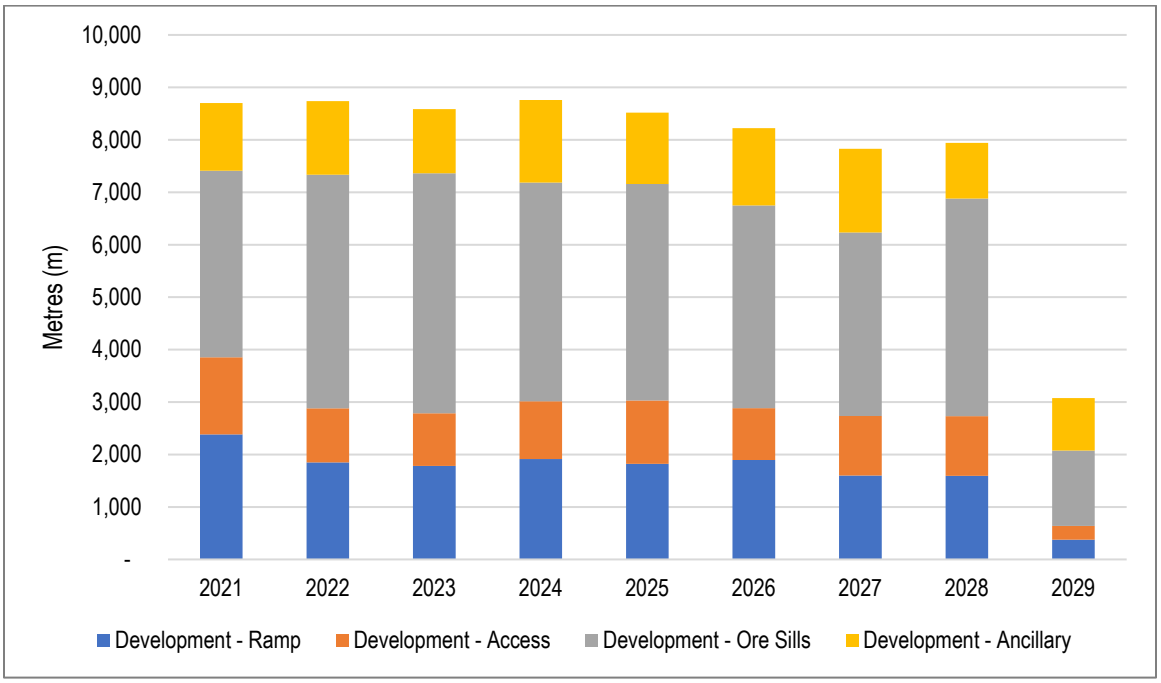


Figure 16-13: Lateral Development by Year

To achieve a production rate of 431,000 t/a (1,200 t/d) a relatively consistent total material movement of approximately 670,000 t/a is required for the period from 2023 through 2028 (Figure 16-14). Continued development of the mine access ramps is critical to production.

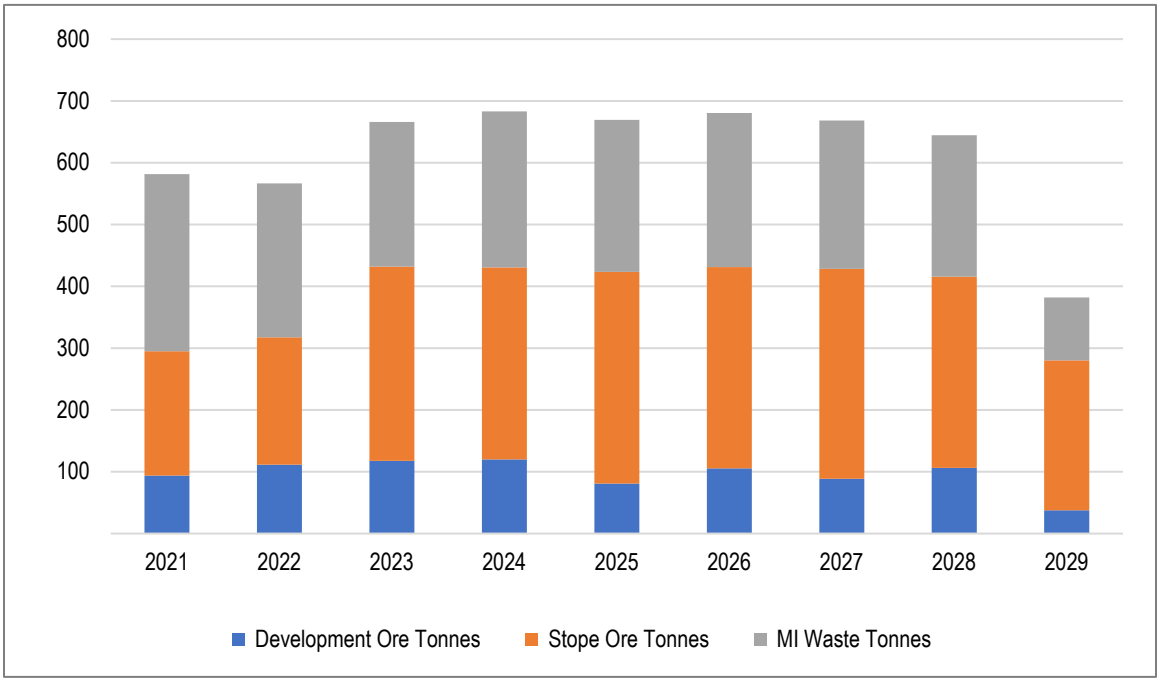
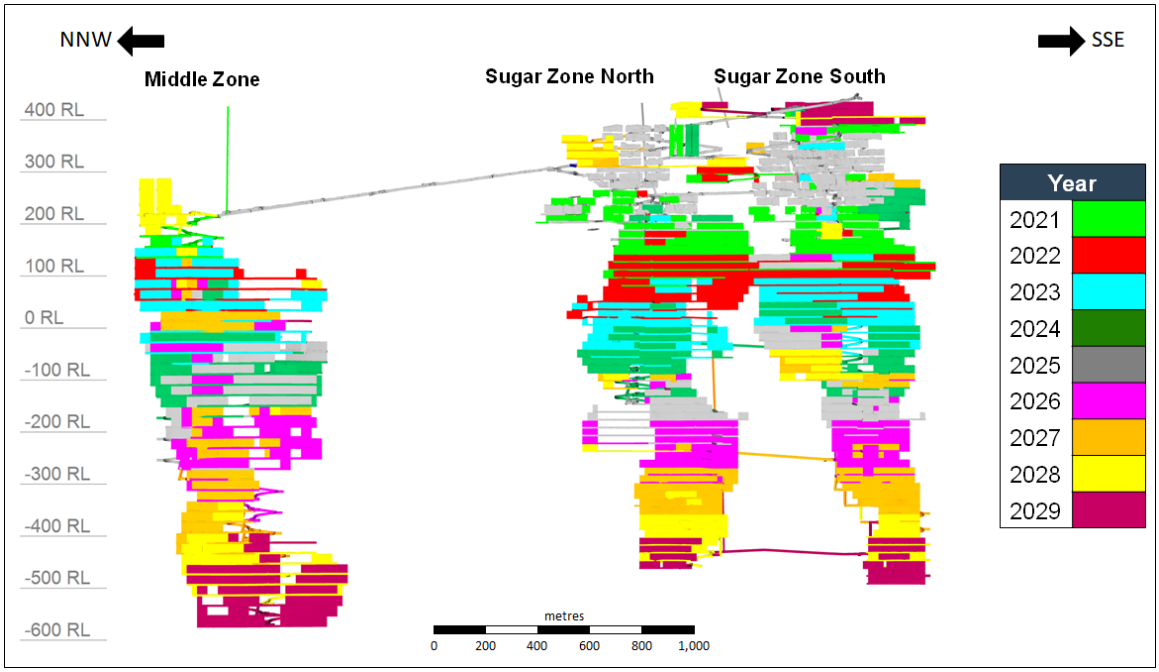


Figure 16-14: Total Material Mined by Year

Table 16-15 and Table 16-10 present the LOM schedule in visual and tabular format, respectively.



Note: The reference elevation at the Sugar Zone Mine is sea level and this is reflected in this figure. Sub-level naming is based on the elevation difference between the portal collar (455 masl) and the sub-level access.

Figure 16-15: Annual Mining Schedule

Table 16-10: Annual Mining Schedule

	Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Development Ore											
Development Ore Tonnes	t '000s	862	94	112	118	120	81	105	89	106	38
Development Ore Grade	g/t	5.52	6.31	6.28	5.43	4.84	6.17	6.52	4.50	4.56	4.57
Development Ore Ounces	oz	152.8	19.1	22.5	20.5	16.7	16.1	22.1	12.8	15.6	5.5
Stoping Ore											
Stope Ore Tonnes	t '000s	2,592	201	206	314	311	342	326	340	309	243
Stope Ore Grade	g/t	7.73	7.52	9.15	8.53	7.71	7.17	9.41	7.75	5.63	6.90
Stope Ore Ounces	oz	644.4	48.6	60.5	86.3	77.1	78.9	98.6	84.6	56.0	53.8
Total Ore											
Total Ore Tonnes	t '000s	3,454	295	317	432	431	423	431	428	416	280
Total Ore Au Grade	g/t	7.18	7.13	8.14	7.69	6.91	6.98	8.71	7.08	5.36	6.59
Total Ore Au Ounces	oz	797.3	67.6	83.0	106.8	95.7	95.0	120.7	97.5	71.6	59.4
Waste											
Waste Tonnes	t '000s	2,088	286	249	234	252	246	249	240	229	102
Total Tonnes	t '000s	5,542	582	567	666	683	669	680	668	644	382
Backfill											
Rockfill Tonnes	t '000s	311	144	139	4	-	-	-	-	4	20
Pastefill Tonnes	t '000s	1,160	-	-	187	182	203	182	161	131	114
Total Backfill Tonnes	**t '000s**	**1,471**	**144**	**139**	**191**	**182**	**203**	**182**	**161**	**135**	**134**

	Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Lateral Development											
Operating Sill Development	m	36,126	3,752	4,668	4,718	4,761	4,512	4,154	3,829	4,242	1,490
Capital Development	m	34,245	4,950	4,070	3,869	3,999	4,005	4,067	4,000	3,699	1,586
Total Lateral Development	**m**	**70,371**	**8,702**	**8,738**	**8,587**	**8,759**	**8,517**	**8,221**	**7,829**	**7,941**	**3,076**

16.15 Waste Backfill

During 2021 and 2022, unconsolidated waste backfill will be used to backfill stopes. This method has been used successfully since the start of stoping operations at the mine. Once a sublevel is complete, waste is delivered by truck from waste development headings to the sublevel truck load-out bay. Then the waste from the truck load-out is transferred into the stope by 1.8 m^3 LHDs. As the muck pile progresses into the stope, ground support rehabilitation is carried out as necessary, and a concrete bumper block chained to the sidewalls is advanced. Stopes are filled at an average rate of 250 t/d.

16.16 Paste Backfill

When the process plant was constructed in 2017/2018 a paste backfill preparation system was installed, consisting of a thickened tailings storage tank; ceramic disc filter unit; cement storage and feed arrangement; high-shear mixer; and an agitated storage hopper. The distribution strategy involved transporting paste by ready-mix truck from the plant to a number of uncased boreholes located to access the primary stoping areas. Significant problems were encountered during the trial operation of the plant in late 2019, and the design of the underground distribution system was found to be flawed. Consequently, paste backfill has not been successfully utilized at the operation, and currently the disc filter system is being used solely to process the tailings stream for the production of filter cake for dry-stacked tailings.

P&C were retained in mid-2020 to prepare feasibility-level designs for preparing and distributing paste backfill to support an initial production rate of 800 t/d, ramping up to 1,200 t/d. During the design process it was determined that a paste plant suitable for an 800 t/d mine production rate was not economically viable due to the low production rate from the current ceramic disc filter, which produced paste at a rate only suitable for an 8 cm underground distribution-line system. The small line-size and the long distances would create a risk of blockages and lead to unsafe distribution-line pressures. To overcome this problem a complex and expensive dry-cake storage and feed system would be required to batch paste in sufficient quantity for a more suitable 10 cm distribution-line system. A value-engineering process determined that this approach was not cost-effective, and a decision was made to continue backfilling with unconsolidated waste until mine production reached 1,200 t/d; at this point a new filtration system with higher throughput would be installed, enabling the paste plant to continuously produce and pour paste backfill through a 10 cm distribution system.

16.16.1 Strength Requirements

Table 16-11 shows the paste backfill-strength design criteria determined by RockEng by RockEng and the estimated binder content from P&C's laboratory test data incorporated in the feasibility study.

Table 16-11: Paste Backfill Strength Requirements

Overhand Mining with 2 m sill pillar between mining blocks		Plug 1-d Cure	Main Fill 3-d Cure	Cap 7-d Cure
North and South Zone (15 m high)	Strength required (kPa)	189	110	442
	Cement content (%)	5.5	2.2	4.3
Middle Zone (20 m high)	Strength required (kPa)	286	173	442
	Cement content (%)	7.4	2.4	4.4

16.16.2 Paste Backfill Quantity and Flow Rate

The backfill requirements are based on the LOM plan and are summarized in Table 16-10.

The upgraded paste plant availability is estimated as 92%/a. The paste plant availability is also dependent on the availability of the process plant and so in combination, the surface plant availability is somewhat lower than 92%.

It is expected that the underground paste distribution system (UDS) availability to receive paste and deliver to the stopes is around 80% when considering, for example, unplanned events, pipe changeovers, ramp blockages, power outages, barricade construction, or no place to pour.

Surface and underground availability are linked, so the overall availability of the system will be less when both factors are considered. Based on the paste plant and UDS availability, the system availability is expected to be approximately 74%.

The utilization rate of a paste plant is generally quite a bit lower than a typical mine process plant. The paste plant buffers the link between the "continuous" process plant operation and the "intermittent" paste filling underground operation. There can be significant downtime associated with backfilling when the availability of the paste plant, the underground distribution, the process plant, and the miners are combined. Extended downtime can be experienced in the case of a plugged pipeline or borehole as well. Extra capacity in the paste plant is therefore vital for catch-up and maintenance.

A typical average paste fill utilization rate of 60% (based on experience in the backfill industry) was assumed when determining the system capacity of the paste plant.

The nominal design flow rate of the paste system is 32 m^3/h (±25%) based on the mine plan. The most optimal paste concentration range is between 76% and 78% solids depending on the paste pour point. This equates to a paste plant capacity of 35 to 60 dry t/h paste.

16.16.3 Testwork Summary

The majority of the tailings consist of the hard silicate minerals (quartz, albite, actinolite, and orthoclase). The carbonate mineral, calcite, is also reported in the tailings' analysis. These minerals are inert and do not participate in the hydration dynamics of the cementitious materials; which makes them good filler materials for backfill applications, however they are known to accelerate wear in pipelines.

To investigate the filtration rates and thickness of the filter cakes, vacuum filtration testwork was carried out over a range of feed-slurry mass concentrations and cycle times. Additional tests were conducted to investigate the use of a potential filter aid (Polyclear 3180C). The increase of feed material pH was also investigated. The filter aid had no significant effect on the testwork. Adjusting the pH of the feed material proved to increase the filtration rate, but the cake moisture content did not improve.

Unconfined compressive strength (UCS) testing was conducted to support determination of the backfill recipe. Various types and quantities of binder were evaluated to determine the recipe that can achieve the specified target strength. The results of the strength testwork are presented in Figure 16-16 and Figure 16-17.

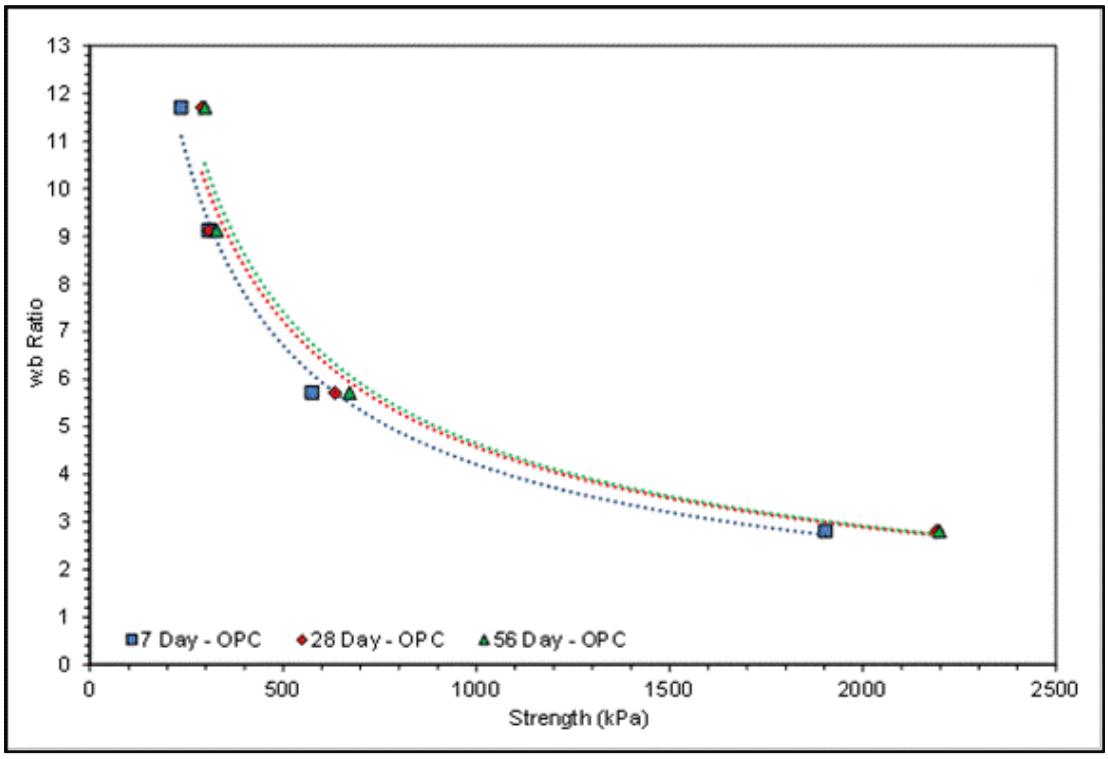


Figure 16-16: Water-to-Binder Curves for OPC Recipes

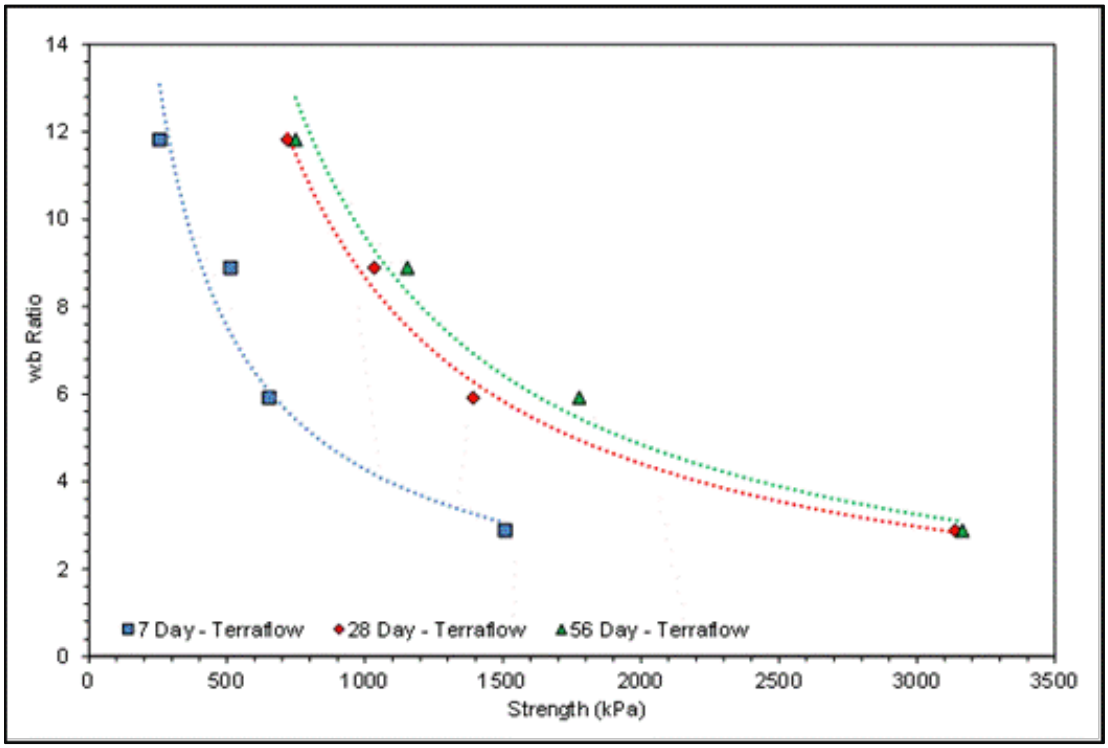


Figure 16-17: Water-to-Binder Curves for Terraflow Recipes

The paste recipes selected for the feasibility study are as follows:

- Vertical Exposures
  - 100–200 kPa
  - 7 d curing time
  - 2.6% binder content (assuming Lafarge Terraflow is used)
  - 76.3% mass concentration.
- Horizontal Exposures
  - 1,250–2,000 kPa
  - 28 d curing time minimum
  - 5.8% binder content (assuming Lafarge Terraflow is used)
  - 76.3% mass concentration.

Horizontal exposures of paste are not currently required by the mine plan due to 2 m sill pillars planned at the top of each third lift within a mining block.

16.16.4 Design Considerations

P&C have completed a high-level investigation into the option of trucking paste to gravity-fed boreholes versus making use of a paste pump and piping system. A paste pumping system was selected for the feasibility study, based on the following conclusions:

- A pumped system will have a higher capital cost but a lower OPEX than a truck-fed gravity system. The payback on the additional cost of the pumped system is approximately three years.
- Pumped systems offer more process flexibility and are more forgiving than gravity-fed systems due to the operational envelope available by adjusting pump output pressure. Gravity-fed systems have no operational envelope at a given flow rate. The increased control afforded by pumped systems allow the operator to keep the main borehole from surface full. In gravity-fed systems it is a challenge for pastefill operators to keep the main borehole from surface full. As such, in a gravity-fed system there is the potential for increased wear in the main borehole. If the borehole casing is damaged enough, through wear, to warrant drilling a new borehole, a significant cost will be incurred.

In deciding the best way forward for the paste plant increased throughput upgrade, the current equipment limitation, tailings availability, and optimal UDS sizing were considered:

- A DN100 (4") UDS was selected based on paste yield stress range, system pressures, and best practices. This results in a tailings requirement of ~44 dry t/h.
- The current tailings production rate is not sufficient to keep a DN100 UDS fully operating (~34 dry t/h available prior to 2023). To increase the tailings feed to the paste mixer, additional tailings will need to be harvested and fed to the mixer via a conveyor-belt system. However, from year 2023 onwards, the mill will have sufficient tailings to keep a DN100 UDS full.
- Once the mill has increased its capacity in 2023 to 1,200 dry t/d, the current ceramic disc filtration system will become the bottleneck of the paste operation with a throughput of ~30 dry t/h. Installing a second ceramic disc filter is not a beneficial option due to the batch nature of ceramic disc filters. During the mandatory acid backwash cycles of the ceramic disc filters the paste plant would stop producing paste and the underground distribution system would need to be flushed clean.
- The two options that were considered to increase the filter cake throughput are:
  - Adding a conveyor-belt system to feed additional tailings to the mixer
  - Replacing the existing ceramic disc filter with a conventional vacuum disc filter.

For the purpose of the feasibility study, the decision was made to replace the existing ceramic disc filter with a conventional vacuum disc filter as it provides additional operational flexibility and reduces risk to the overall paste backfill distribution system.

16.16.5 Paste Plant Layout

Figure 16-18 is a simplified process flow sheet of the current paste plant and illustrates the basic system components.

All flotation tailings from the mill are pumped to the tailings thickener. The underflow from the tailings thickener is pumped to an agitated filter feed tank from which slurry is pumped to a ceramic disc filter via a splitter box. The filter feed splitter box divides slurry between the ceramic disc filter, paste mixer, and filter feed-tank.

The ceramic disc-filter cake reports to the paste mixer via the filter cake belt. Paste is discharged from the paste mixer into a live bottom hopper (with screw feeder/agitator).

The cement system consists of a storage silo, which is filled pneumatically by a bulk tanker, and a weigh bin. The weigh bin is fed from the storage silo via rotary feeder and air-blow system.

In the current design, the live bottom hopper is equipped to discharge into a ready-mix truck for transport to distant boreholes.

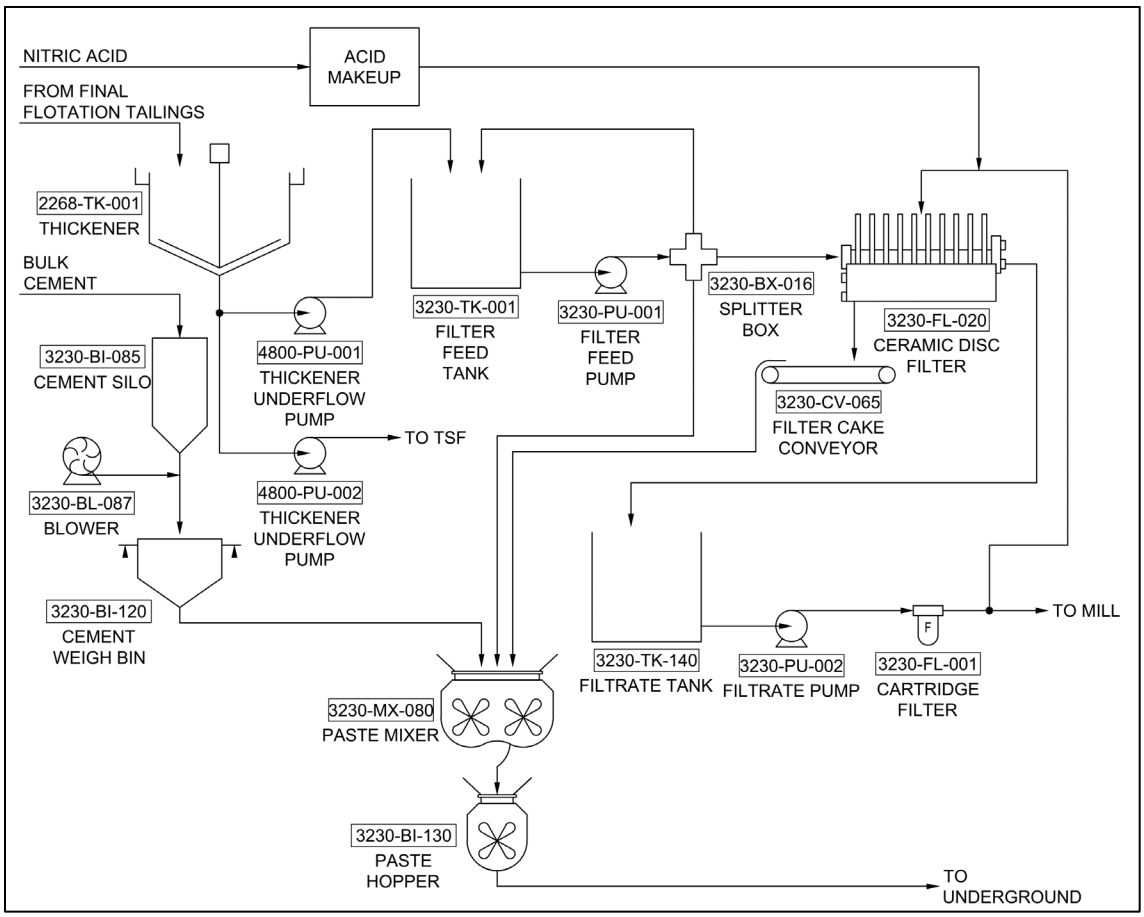


Figure 16-18: Current Paste Plant Flowsheet

The changes made to the current system to sustain the increased throughput are as follows and represented in Figure 16-19:

- Replace the existing ceramic disc filter with a conventional vacuum disc filter
- Replace the current truck load-out system with a hydraulic-piston paste-pump system
- Add additional compressed air capacity
- Replace the current cement weigh blower (3230-BL-087) with larger-capacity equipment
- Install a DN100 paste reticulation system
- Install an underground booster station as part of the pumped UDS.

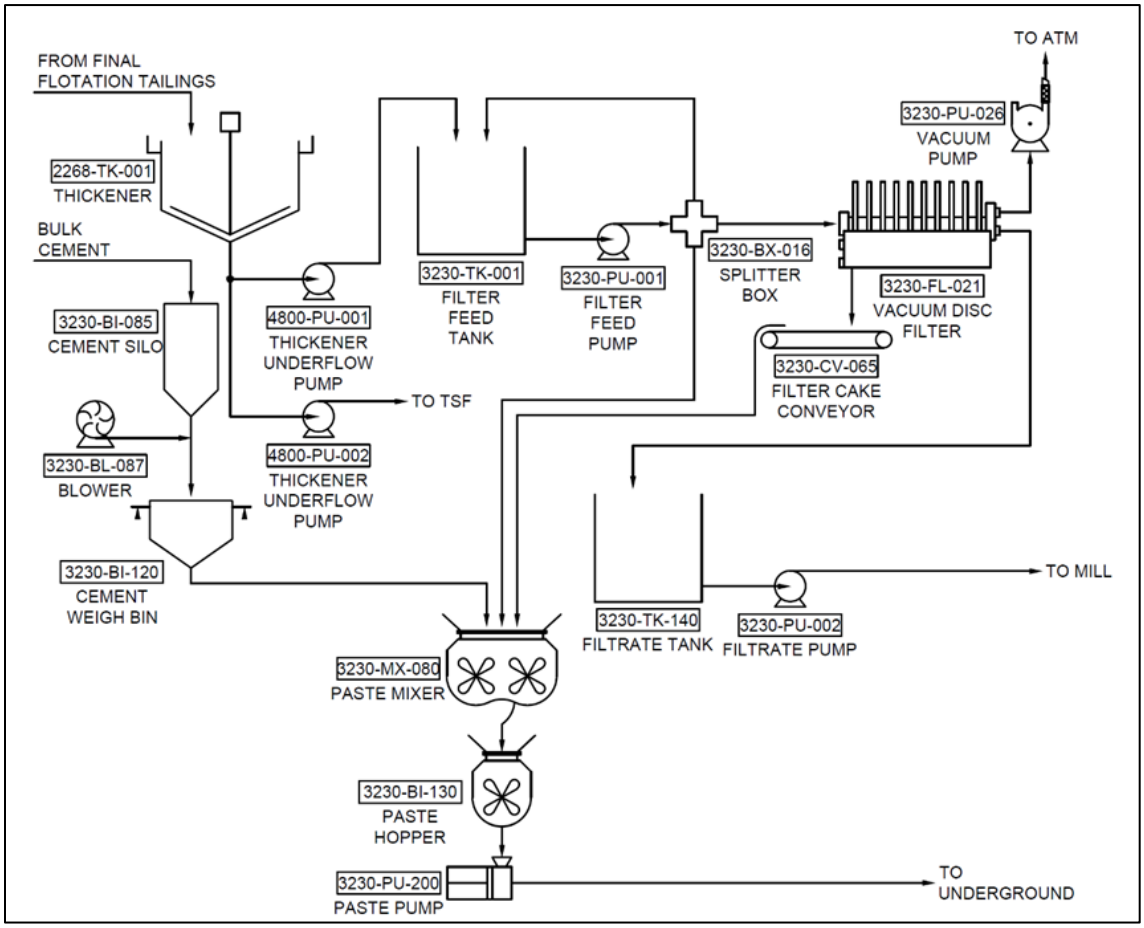


Figure 16-19: Future Paste Plant Flowsheet

16.16.6 Underground Distribution

P&C conducted a steady-state hydraulic analysis of the Sugar Zone UDS (which include the North, South, and Middle Zones) using rheological properties and the pastefill design flow rate discussed previously. The Sugar Zone paste fill UDS is designed as a pumped system to reach all the required underground stopes in the North, South, and Middle Zones. The following summarizes the hydraulic analysis assumptions and outputs:

- Borehole routing was completed in consideration of the mine plan and available level access through the LOM.
- The UDS has been designed to operate with pastefill within a wide flow-rate range of 24 to 40 m^3/h.
- DN 100 piping provides a suitable flow velocity range of 0.88 m/s to 1.47 m/s for the paste backfill system. Smaller diameter piping would add blockage risk, increase the friction losses, and excessive pressure would be required to distribute the paste throughout the reticulation system.
- Due to the long lengths of horizontal piping needed to reach the South and North Zone stopes, a hydraulic piston pump is required at the paste plant. P&C recommended a pump rated for 150 bar (nominal operating discharge pressure of 135 bar) to achieve a pastefill with a yield-stress range of 240 to 340 Pa when pumping to the North and South Zone stopes.
- In addition, the Middle Zone will require a 150-bar hydraulic piston pump within an underground booster station due to the long transfer distances, and will be capable of producing an operating yield stress range of 230 to 280 Pa when pumping to the Middle Zone stopes.

Figure 16-19 shows the paste underground distribution layout with boreholes and pipes shown in blue. Duplicate boreholes are shown in some areas to provide redundancy and manage risk.

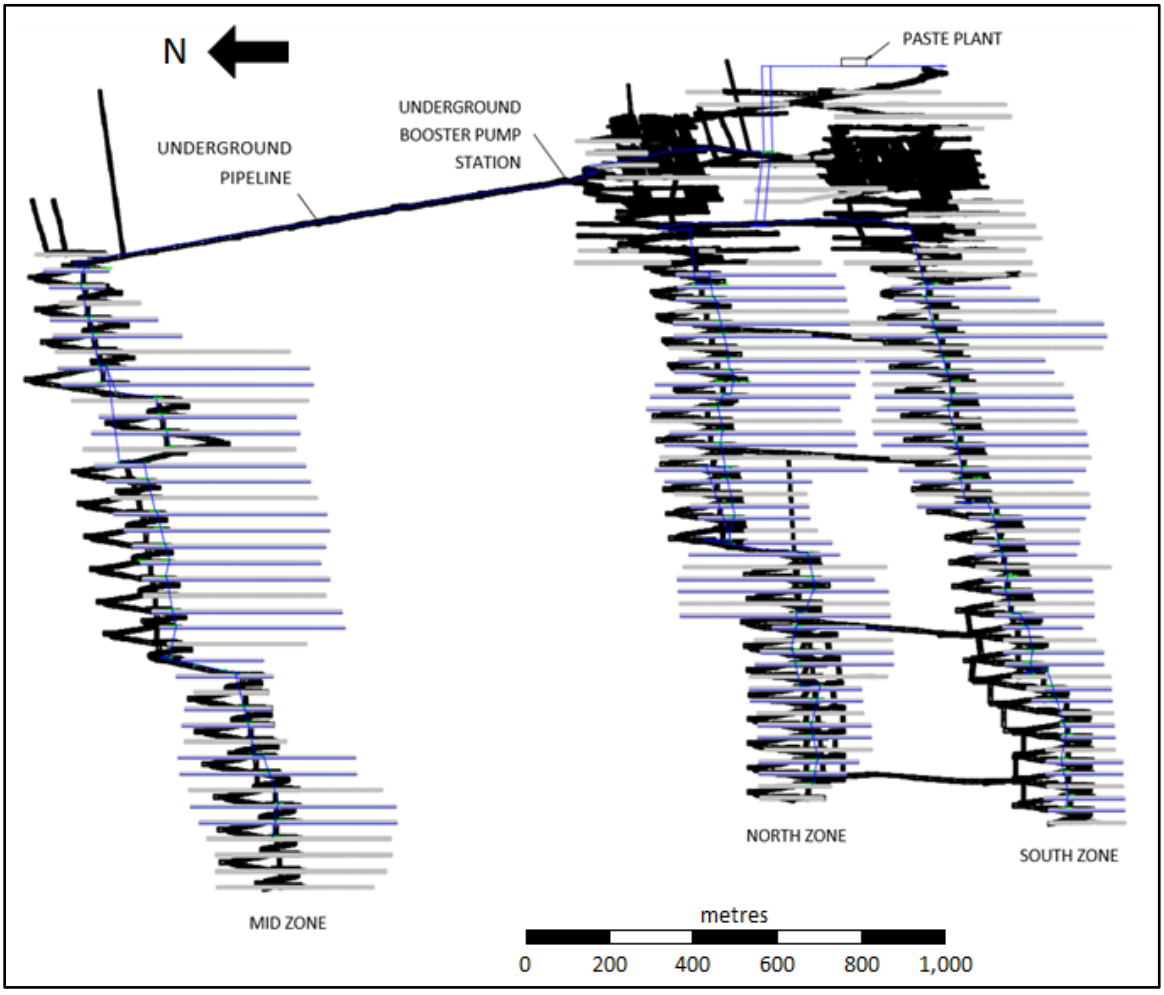


Figure 16-20: Paste Underground Distribution System Layout

16.16.7 Paste Barricade Construction

RockEng has analytically and empirically evaluated the cemented paste backfill strength requirements for meeting the open stoping demands at the Harte Gold Sugar Zone Mine. These strength requirements are based on the vertical paste exposures, undercut paste exposures, and following three types of pours: plug, bulk (main), and cap. These strengths have been provided for expected stope widths, as shown in Table 16-11.

RockEng has analytically and empirically evaluated the requirement for waste rock barricades (WRB) and arch shotcrete barricades at Sugar Zone Mine. The WRB are not engineered, as no engineering controls on the material to build the barricade will be used (i.e., no control over grain-size distribution or compaction). Any risk with the associated variations in design of the WRB can be controlled by observation as this barricade system will leak prior to failure, and the pour can be halted if a leak is observed. The WRB should initially be constructed using a compacted starter berm located coincident with the brow, and the final WRB geometry should extend back into the undercut from this starter

berm with compacted side slopes at 1:1 (45°), and the crest (top) of the WRB in contact with the undercut's back (roof) for a crest length at least 3 m. This length and tightness (compaction level) of crest is the most critical part of the WRB construction. QC procedures should be put into place to ensure that this is being maintained.

Arch shotcrete barricades for Harte Gold will vary: 20, 25, and 30 cm for 4.5, 5.3, and 6.8 spans of the barricade respectively. This thickness assumes that the barricade formwork uses a prefabricated form that can provide a consistent and accurate arch. If a built-in-place design is used, then the minimum thickness of the shotcrete increases from 20 to 25 cm.

16.17 Mine Ventilation

16.17.1 Overview

Harte commissioned JMVL to prepare a ventilation design to support an increase in the mining rate from 800 t/d to 1,200 t/d. This section summarizes the airflow requirement estimates, primary ventilation plan, and ventilation infrastructure and controls required for the long-term design. JMVL estimated the size and location of primary ventilation fans and air heaters, appropriate raise sizing, and ventilation controls such bulkheads, doors, and regulators. JMVL also estimated the capital and steady-state OPEX of the ventilation system.

The airflows required are driven by regulated diesel emission requirements, best-practice velocity requirements for air circulation, and dust control requirements.

16.17.2 Mine Ventilation Layout

The Sugar Zone main access ramps will act as the primary air intakes, and the exhaust vent raises will extend vertically between mine levels and be developed concurrently with each ramp. Propane gas-fired heating systems will be used during the winter months to heat the mine intake air.

The Middle Zone will have an independent exhaust raise to surface. The primary ventilation layout is provided in Figure 16-20.

Secondary ventilation will comprise axial flow fans and flexible vent ducting in all drifts without primary airflow.

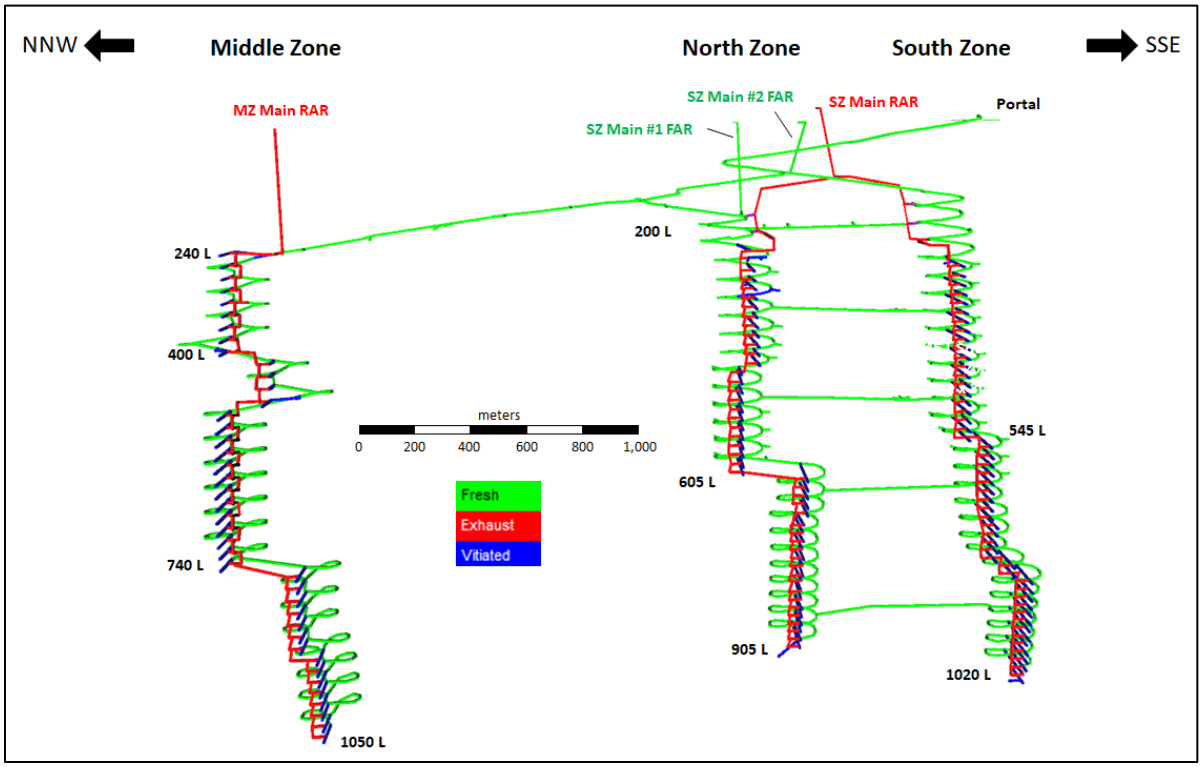


Figure 16-21: Longitudinal View of Primary Ventilation Circuit

16.17.3 Airflow Requirements

The estimated underground air volume requirements have been based on the Ontario *Occupational Health and Safety Act*, Regulations 854 Section 183.1 (3). The regulation states: *"*The flow of air must be at least 0.06 m^3/s for each kilowatt of power of the diesel-powered equipment operating in the workplace*."*

The airflow requirement is commonly expressed in cubic feet per minute (cfm) and the conversion is 100 cfm per horsepower (hp).

The maximum ventilation demand would occur if all mobile equipment in the mine was operating simultaneously. Although this is possible, in practice it is very unlikely to occur. To estimate a more likely peak ventilation demand, utilization factors have been applied to the mobile equipment. The utilization factors reflect the likely combination of equipment that will be running during the busiest periods of any working shift.

Utilization factors vary with the type of equipment, and reasonable judgement has been used. Equipment such as drill jumbos, which operate on diesel power only while moving from one workplace to the next, are utilized less than LHDs or haul trucks, which work continuously throughout the shift. Ventilation simulation modelling and associated facility designs have been prepared to reflect the expected equipment utilizations.

The capacity required for each stage is related to the peak level of development and production activity. The equipment horsepower, utilization factors applied to the equipment fleet, and resulting ventilation flows at steady state operation are shown in Table 16-12.

Table 16-12: Underground Diesel Equipment Fleet Steady-state Air Volume Requirements

Unit	Quantity	HP (each)	kW (each)	Diesel Engine Utilization (%)	Total Hp	Total kW	Total CFM (100 cfm/hp)	Total (m^3/s)
Ramp Development (4.5 m x 4.3 m)								
Two-Boom Jumbo	1	78	58	25	20	15	1,950	1
4.5 m^3 LHD	1	270	201	100	270	201	27,000	13
Scissor Lift	1	147	110	25	37	27	3,675	2
Toyota Jeep	1	128	95	50	64	48	6,400	3
Subtotal	-	-	-	-	-	-	39,025	18
Production / Ore Development (2.8 m x 3 m)								
Single Boom Jumbo	1	65	48	25	16	12	1,625	1
1.8 m^3 LHD	2	129	96	100	258	192	25,800	12
Subtotal	-	-	-	-	-	-	27,425	13
Ramp Hauling and Miscellaneous								
30 t Truck	2	415	309	100	830	619	83,000	39
Toyota Jeep	1	99	74	50	50	37	4,950	2
Boom Truck	1	147	110	25	37	27	3,675	2
Subtotal	-	-	-	-	-	-	91,625	43
Summary	-	-	-	-	-	-		
Ramp Development	-	-	-	100	-	-	39,025	18
Production	-	-	-	100	-	-	27,425	13
Ramp	-	-	-	100	-	-	91,625	43
Subtotal	-	-	-		-	-	158,075	75
Leakage (Modelled)	-	-	-	-	-	-	36,357	17
Subtotal UG Density 1020 Level FA Required per Area						0.081	194,432	92
Subtotal Surface Density FA Required per Area						0.075	209,987	99
Airflow from Fresh Air Raise to Portal (Allow for 1 truck @ 41,500 cfm)							41,500	20
Total UG Density FA Required All Zones (3)							**624,797**	**295**
Total Surface Fan Volume FA Required All Zones (3)							**671,460**	**317**
Total							**671,460**	**317**

16.17.4 Ventilation Modelling

JMVL performed ventilation simulation modelling, and associated infrastructure designs were prepared based on the following:

- Mine development and production plans, including access and production level 3-D designs
- Mine production schedules
- Airflow requirements as calculated from mobile equipment list and engine power estimates.

The ventilation models have been prepared to simulate the airflows during each of the stages of the ventilation infrastructure upgrade and steady-state (full production), where mining is active in most areas and the ventilation demand is at peak.

Isometric views of the ventilation numerical models by stage have been presented for illustration purposes in Figure 16-22 through Figure 16-28 (not to scale), with red showing exhaust air, blue showing mixed air, and green showing fresh air (FA).

Stage 1—Q4 2020 Commissioning # 1 Main Fresh Air

- Surface # 1 FA fan and heater installed and commissioned
- Regulator installed bottom of FA raise (FAR).

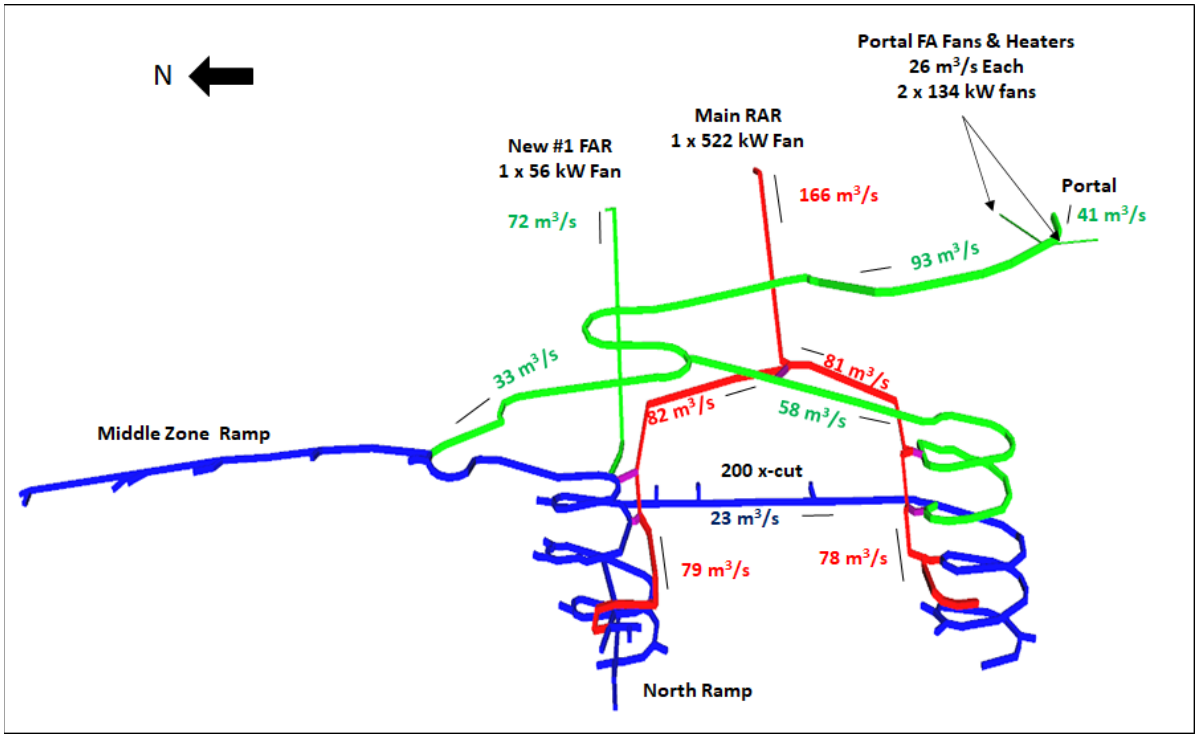


Figure 16-22: Stage 1 Q4 2020 Commissioning #1 FAR (Isometric View, Not to Scale)

Stage 2—Q3 2021 Commissioning Middle Return-Air-Raise

- Complete development of Middle return-air-raise (RAR) to surface and installation of manway
- Installation of Middle Zone 240 L # 1 underground RA booster fan.

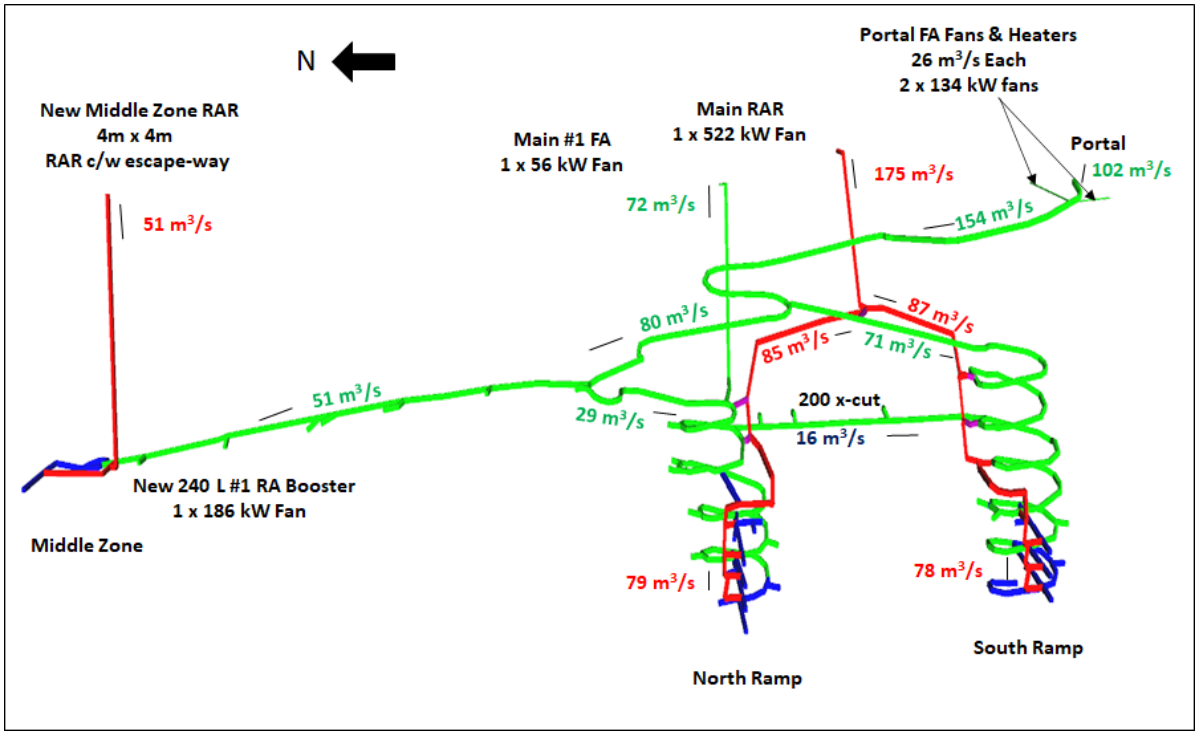


Figure 16-23: Stage 2 Q3 2021 Commissioning Middle Zone RAR (Isometric View, Not to Scale)

Stage 3—Q4 2021 Production Middle Zone

- First production from Middle Zone.

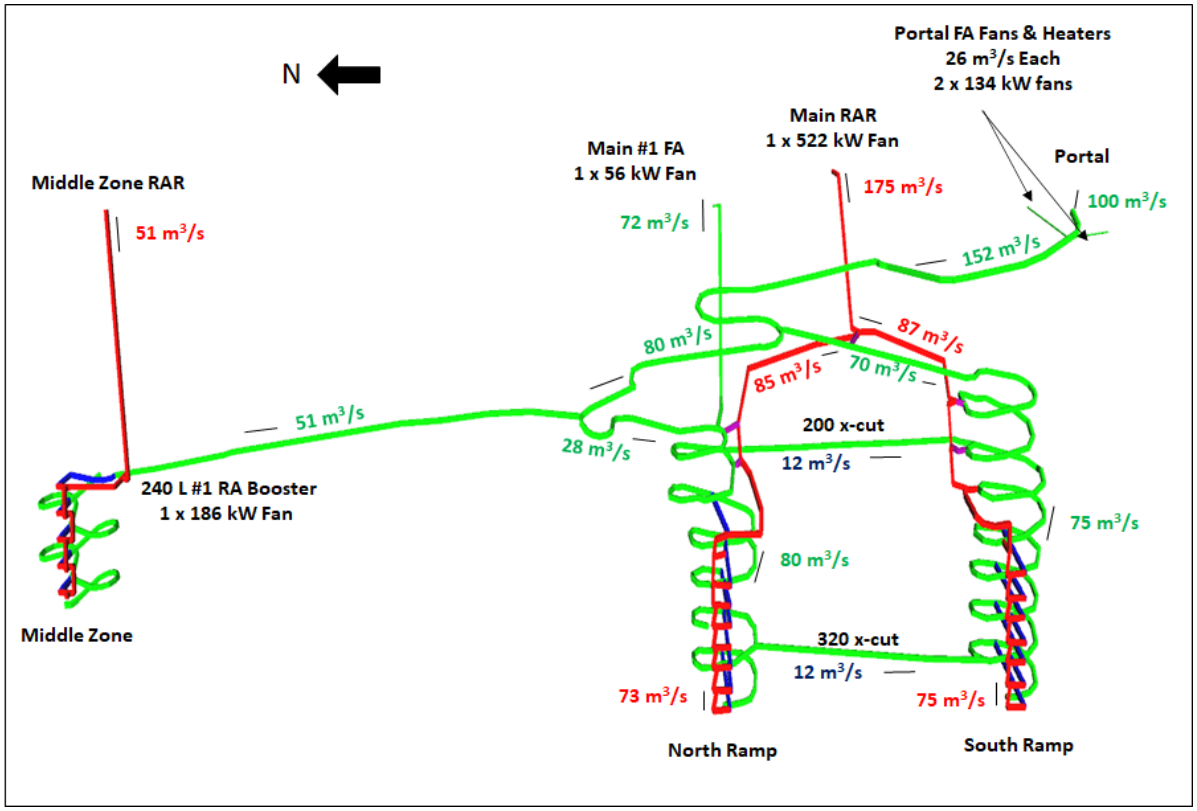


Figure 16-24: Stage 3 Q4 2021 Production Middle Zone (Isometric View, Not to Scale)

Stage 4—Q3 2022 Commissioning #2 Main Fresh Air

- Complete development of #2 FAR to surface
- Surface # 2 FA fan and heater installed and commissioned
- Surface # 2 RA fan installed and commissioned.

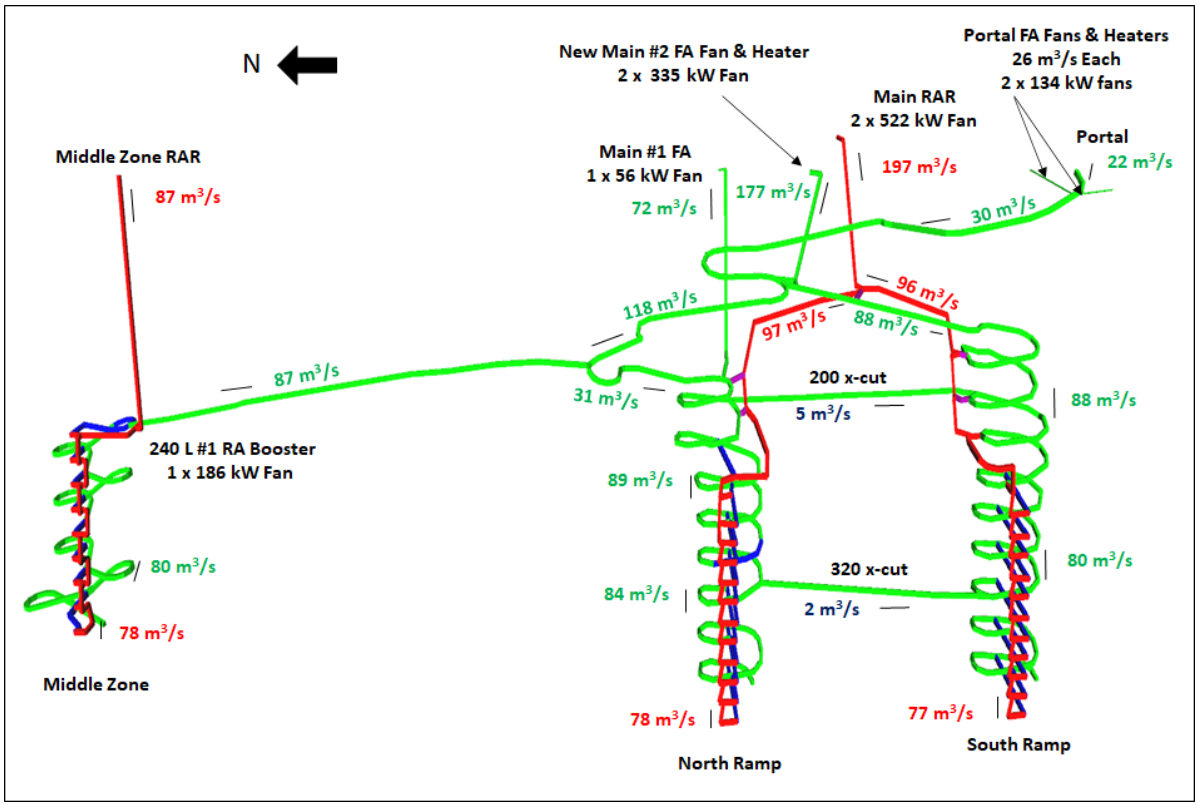


Figure 16-25: Stage 4 Q3-2022 Commissioning #2 FAR (Isometric view, Not to Scale)

Stage 5—3rd X-Cut North—South Ramps

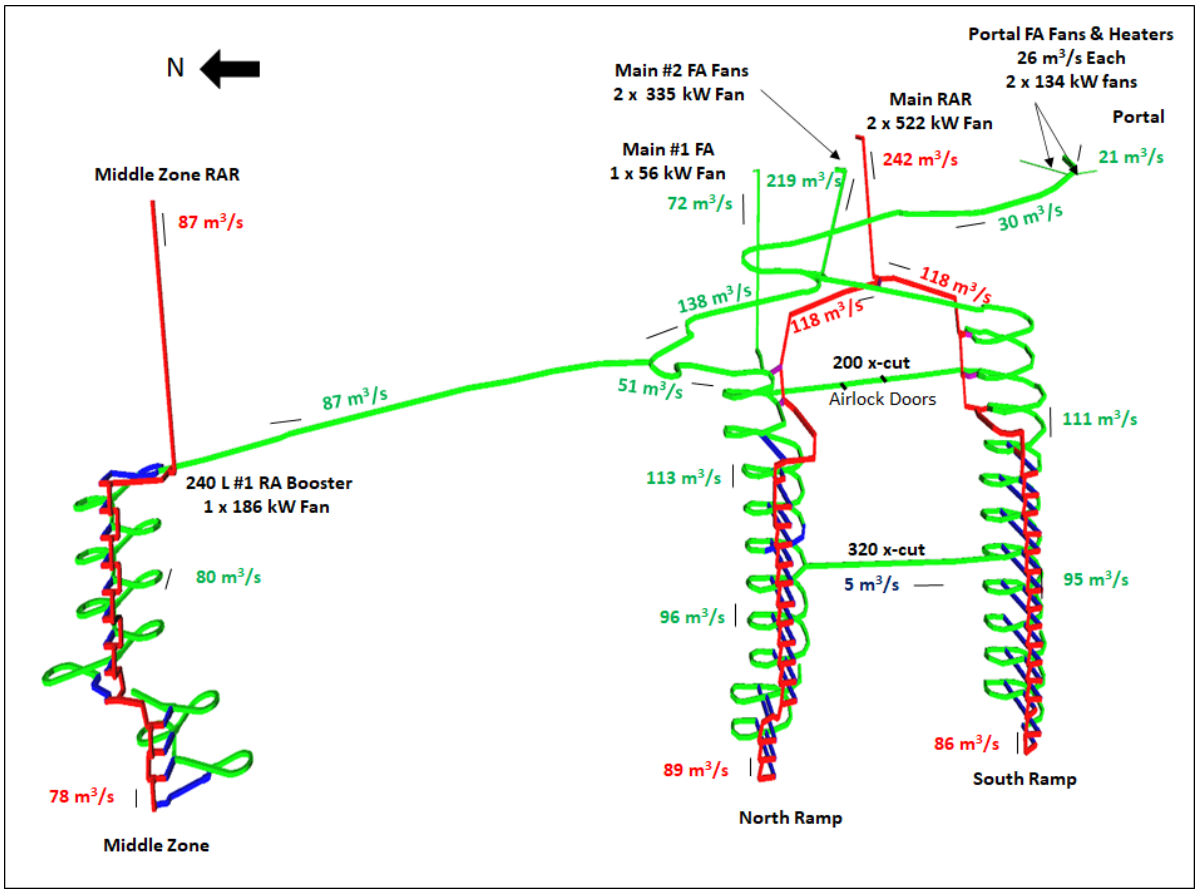


Figure 16-26: Stage 5—3rd X-Cut North–South Ramps (Isometric View, Not to Scale)

Stage 6—4th X-Cut North—South Ramps

- Installation completed, Middle Zone 400 Level # 2 underground RA Booster Fan
- Installation completed, North 605 Level and South 545 L RA Booster Fans.

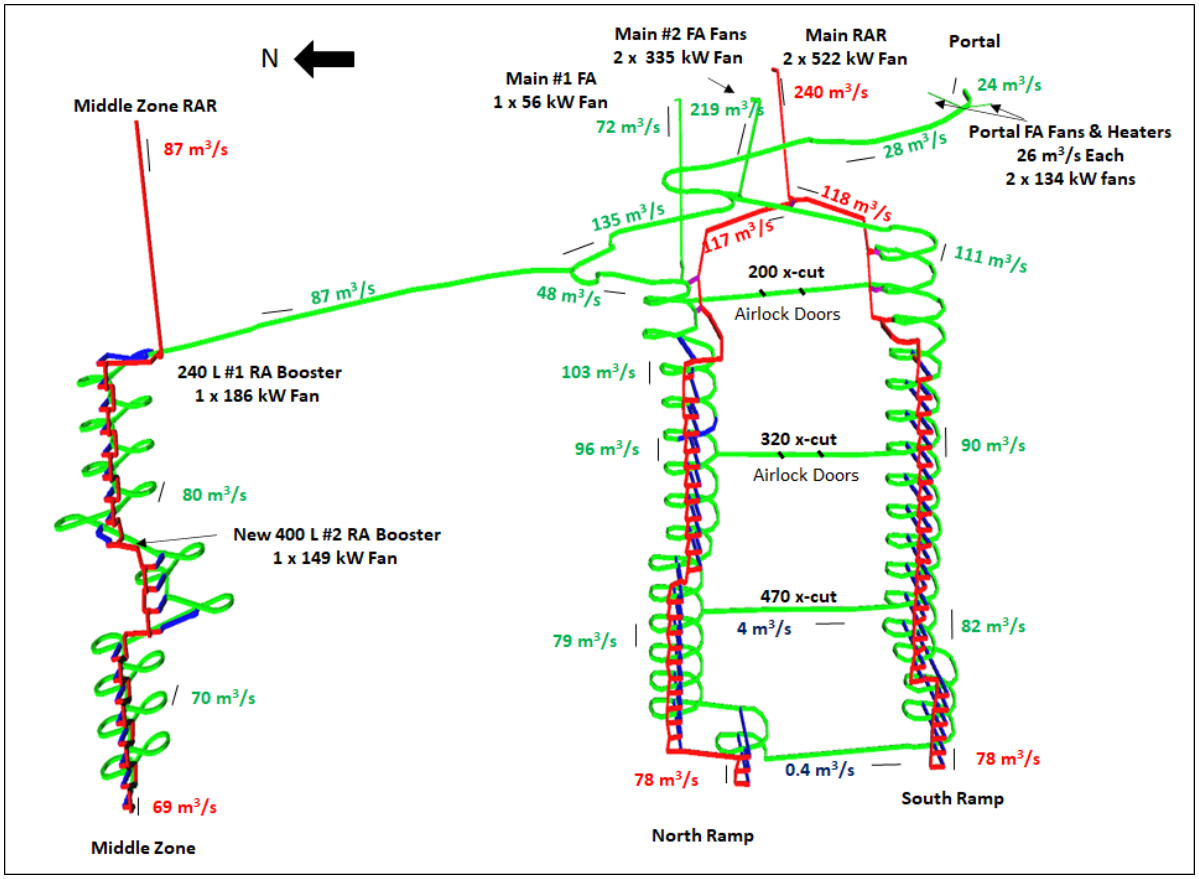


Figure 16-27: Stage 6—4th X-Cut North–South Ramps (Isometric View, Not to Scale)

Stage 7—Life-of-Mine Steady State

- Installation completed, Middle Zone 740 Level # 3 underground RA booster fan.

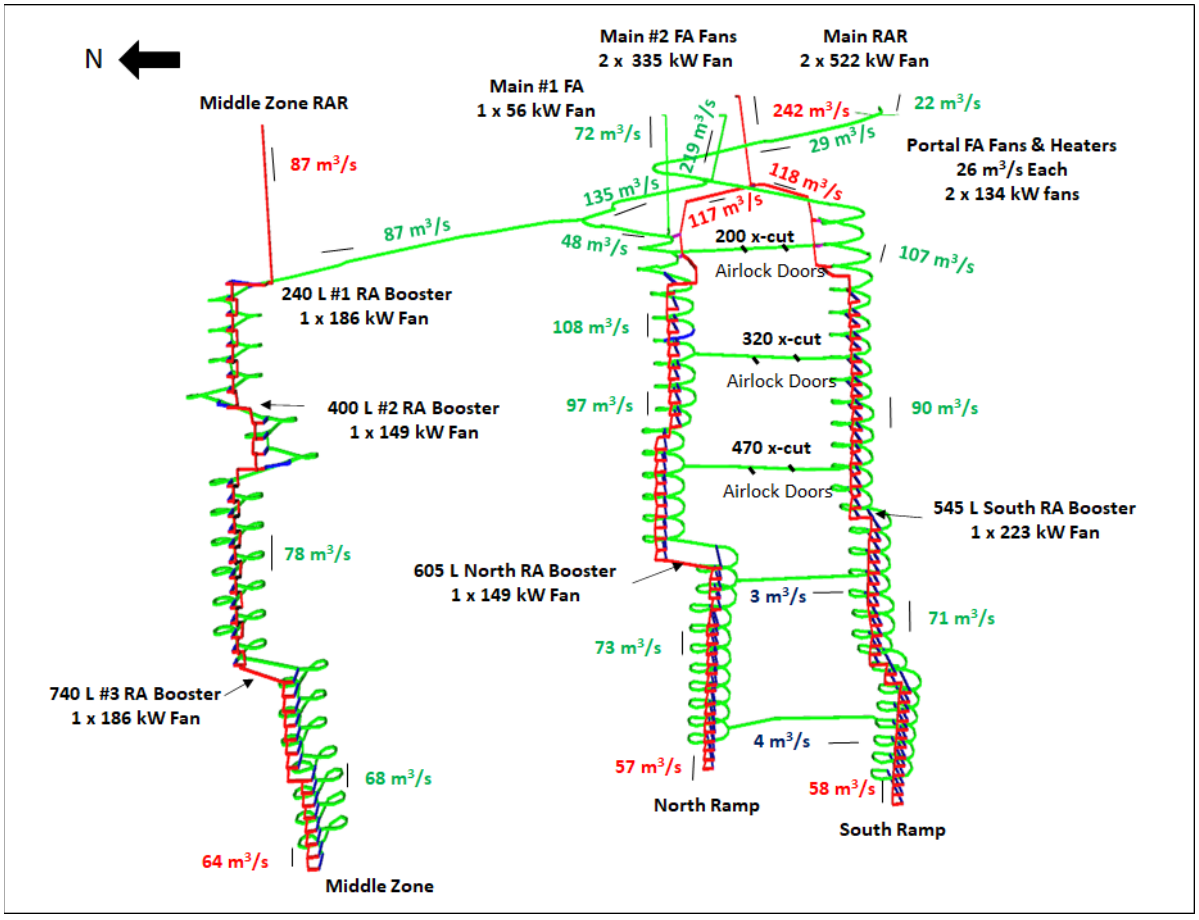


Figure 16-28: Stage 7 Steady State (Isometric View, Not to Scale)

16.17.5 Fan Specification

Main fan operating duties for the modelled scenarios and the resultant maximum duties used for fan specification are shown in Table 16-13.

Table 16-13: Fan Specifications

Fan Location	Qty.	CFM	Ps (Out)	Pt (Out)	Density	Fan Model	RPM	Eff (%)	Each Fan (HP)	Total (HP)	Total (kW)
Surface #2 Fresh Air Fan and Heater	2	232,500	1.3	2.0	0.072	8400 VAX 3150	1,180	71.0	250	500	373
Middle Zone #1 UG Return Aire Booster	1	185,000	3.1	3.70	0.069	7800 VAX 3150	1,180	82.0	250	250	186
Middle Zone #2 UG Return Air Boosters	1	172,000	1.6	2.10	0.070	7800 VAX 3150	1,180	77.6	200	200	149
Middle Zone #3 UG Return Air Boosters	1	170,000	2.6	3.10	0.076	7800 VAX 3150	1,180	77.6	250	250	186
North Zone UG Ramp Return Booster	1	168,000	2.4	2.90	0.071	7800 VAX 3150	1,180	77.8	200	200	149
South Zone UG Ramp Return Air Booster	1	200,000	3.3	4.00	0.070	7800 VAX 3150	1,180	79.2	300	300	224
Auxiliary Level Fans	17	43,000	-	5.00	0.075	42-27-1780	1,800	-	100	1,700	1,268
Total Installed HP										**3,400**	**2,535**

Notes: Ps = static pressure; Pt = total pressure.

16.18 Mine Services

16.18.1 Electrical

16.18.1.1 Current System

Power to site is provided be a 45 km, 25 kV overhead line along Highway 631 north from the town of White River. This power line is connected to the site substation and is transformed from 25 kV to 13.8 kV and 4.16 kV by two transformers with 5 MVA capacity. Each transformer provides power to two separate switchgear lineups (Stantec, 2021).

The 13.8 kV switchgear lineup feeds:

- Portal Infrastructure:
  - Air compressors
  - Ventilation fans (upper ramp)
- Underground Infrastructure:
  - South Zone ramp (to 245 Level)
  - North Zone ramp (to 245 Level)
- Surface Infrastructure:
  - Admin/office facility, dry facilities, warehouse, and coreshack
  - Maintenance facility
  - Security/guardhouse
  - First-aid facility
  - Roadway lighting.

The 4.16 kV switchgear lineup feeds:

- Mill Infrastructure:
  - Mill facility
  - Ore crushing
  - Concentrator
  - Effluent treatment plant
  - Paste plant
  - Sewage treatment plant
- Surface Ventilation:
  - Main fresh air fans
  - Main return air fans.

Underground, the power distribution system originates from the surface substation E-house. A 13.8 kV circuit runs along an overhead power line from the substation to the mine portal and continues down the ramp to various electrical rooms for local distribution. At the 95 Level electrical cut-out, a 3-cell 15 kV line-up splits power between the North Zone and South Zone ramps. At the 140 Level in the North Zone ramp, the electrical substation (ESS) allows for power to split again to provide power to the Middle Zone ramp.

Power continues to be distributed down the north and south ramps to the 245 Level; development work is progressing to access the 260 Level.

Table 16-14 shows the current loading on the 13.8 kV and 4.16 kV switchgear.

Table 16-14: Current Switchgear Loading

Voltage Rating	Area	Connected Load (kVA)	Demand Load (kVA)	Operating Load (kVA)	Main Power Loads
13.8 kV	UG Loads (WBS 2000)	1,676	1,418	874	Equipment from Portal to 245 Level (level fans, pumps, mining equipment, shop, etc.)
	Surface Infrastructure (WBS 3000)	1,008	859	667	Surface air compressors, shop, guardhouse, first aid, road lighting, and warehouse/coreshack/office/dry
	Total	**2,684**	**2,277**	**1,541**	
	Transformer Capacity			5,000	
				30.8%	
4.16 kV	Mill Complex (WBS 4000)	3,004	2,497	1,852	Mill facility, surface crushing, concentrator, ETP, paste plant, sewage treatment plant
	Surface Infrastructure (WBS 3000)	1,994	1,694	1,494	Fan facilities (fresh/return)
	Total	**4,998**	**4,191**	**3,346**	
	Transformer Capacity			5,000	
				67.0%	

16.18.1.2 Upgrade Requirements

The mine design for the planned extension, the North and South Zones will continue from the current 245 Level to reduce cabling for the mine extension, the fresh-air escapeways will be utilized for cable routing. This will continue for the current 13.8 kV circuit from the 260 Level to 335 Level for the north ramp and to 350 Level for the south ramp. As shown in Table 16-14, the 13.8 kV circuit is approximately 45.5% used based on demand and 30.8% operating. As development continues, the power draw on this circuit will start to lessen, allowing more power to be available to the Middle Zone area.

At 350 Level a main cross-cut drift will be developed, and a second circuit will be run from the surface substation down the fresh-air raise to a primary electrical room (Figure 16-28) containing a small switchgear with a main-tie-main incoming configuration for the new circuit and the continuation of the south ramp circuit from 335 Level. This allows power flexibility and redundancy to cater for an upset condition.

From 350 Level, 13.8 kV feeders will run down the north and south fresh-air escapeways to provide power for level infrastructure and development/production mining crews. Each of the three zones has about 40 levels, and power will be distributed in blocks of five levels with the sequence of installation shown in Figure 16-29. As development continues down each area, a mine power centre (MPC) will be connected to the 15 kV load break switch installed in the level electrical rooms (shown for 365 Level, 380 Level, and 410 Level) for development and production activities.

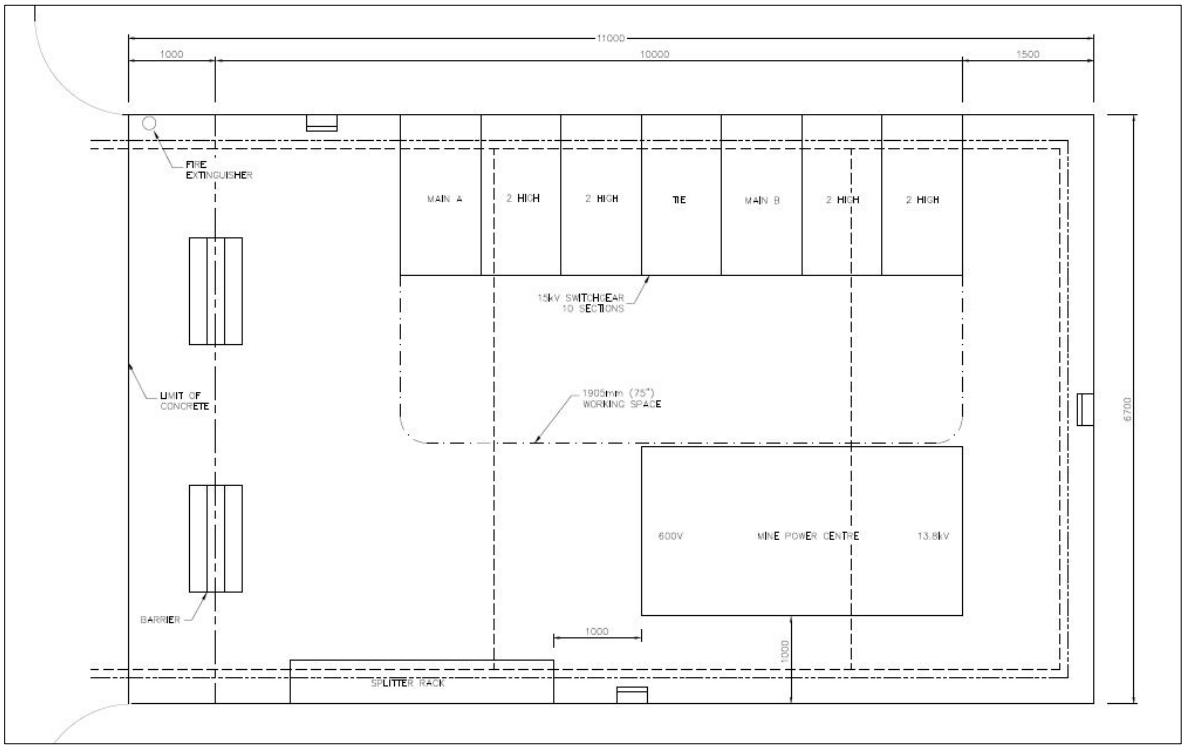


Figure 16-29: Primary Electrical Room Layout

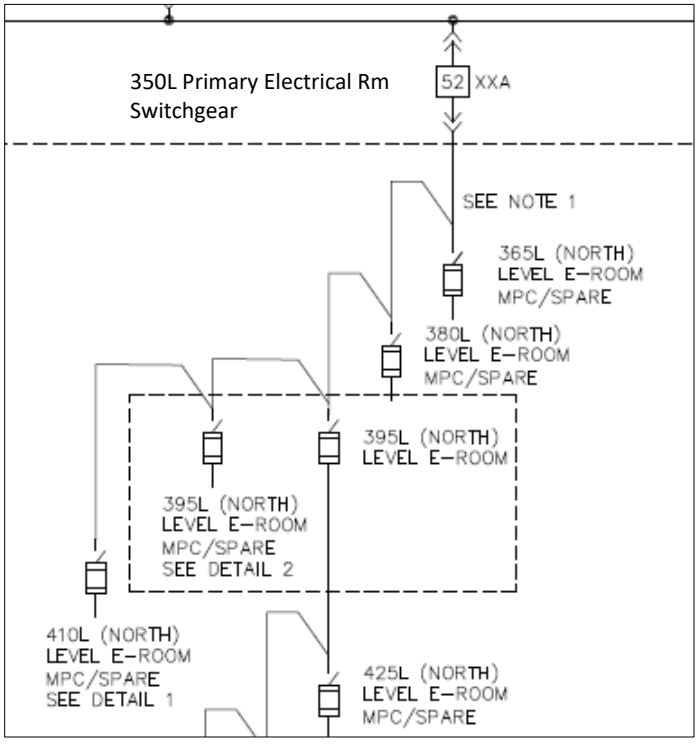


Figure 16-30: Mining Block Single Line

For the 395 Level electrical room, the MPC will be left in this room to provide 600 V power up and down to the other levels within the mining block for infrastructure loads (ventilation system fans, dewatering system pumps, and other facilities like refuge stations and process water stations).

Each of the three zones will have eight mining blocks, so a total of 24 MPCs will be phased in as required until the mine is complete.

Secondary electrical rooms (Figure 16-29) and level electrical rooms are the same except for the second 15 kV load break switch (Figure 16-30). This second switch provides power to the next mining block and is located in the room where the MPC is permanently installed.

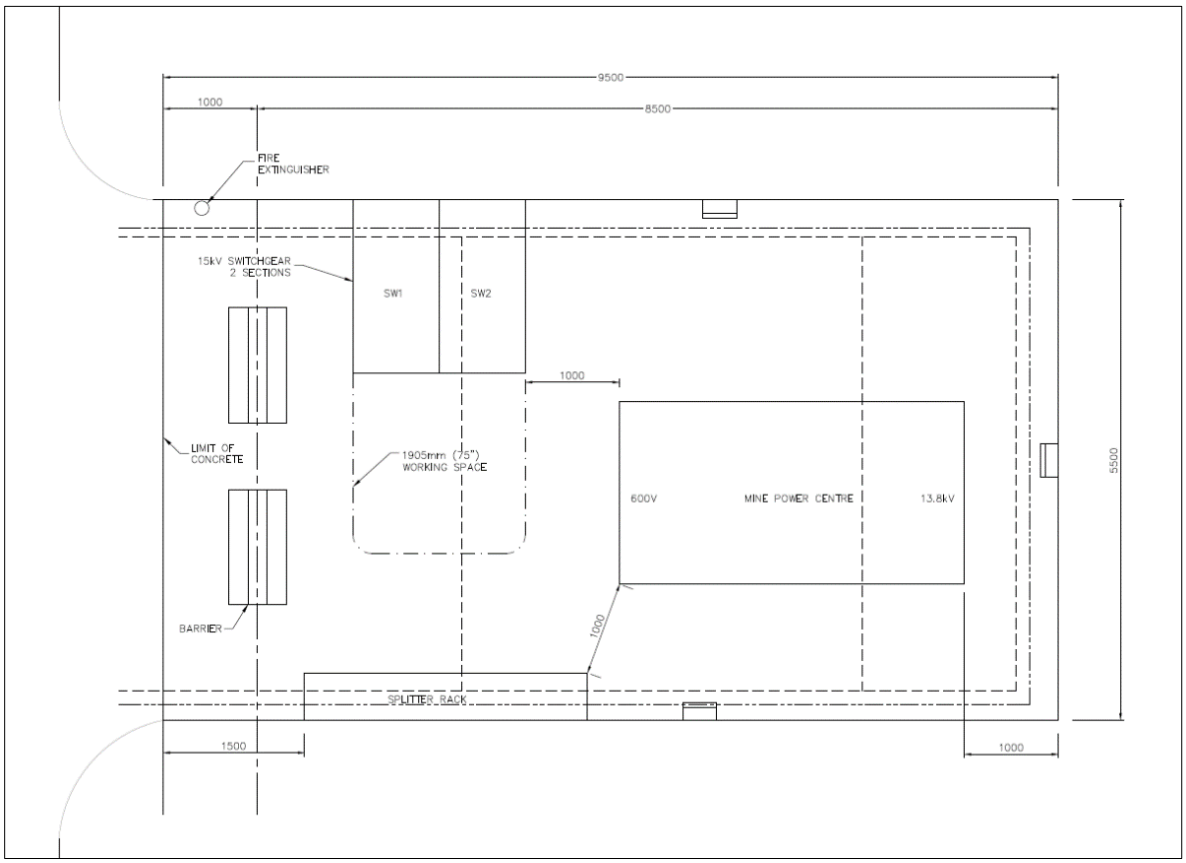


Figure 16-31: Secondary Electrical Room Layout

As development and production move from level to level, power loading will increase as minimal level infrastructure is required. Table 16-15 is based on typical level infrastructure loads and specific area's loads.

Table 16-15 shows the future loading on the 13.8 kV and 4.16 kV switchgear.

Table 16-15: Future Switchgear Loading

Voltage Rating	Area	Connected Load kVA	Demand Load kVA	Operating Load kVA	Main Power Loads
13.8 kV	Existing UG Loads (WBS 2000)	1,676	1,418	874	Equipment from Portal to 245 Level (level fans, pumps, mining equipment, shop, etc.)
	Typical Block Infrastructure	226	192	140	Anticipated loads for 350 Level to mine bottom for all areas Power for North Zone is provided by South Zone to provide flexibility with MZ.
	South Zone Loads	1,361	1,157	663	
	North Zone Loads	141	119	105	
	Middle Zone Loads	1,235	1,050	582	
	Surface Infrastructure (WBS 3000)	1,008	859	667	Surface air compressors, shop, guardhouse, first aid, road lighting, and warehouse/coreshack/office/dry
	Total	**5,647**	**4,795**	**3,031**	
	Transformer Capacity			5,000	
				60.6%	
4.16 kV	Mill Complex (WBS 4000)	3,004	2,497	1,852	Mill facility, Surface Crushing, Concentrator, effluent treatment plant (ETP), Paste Plant, Sewage Treatment Plant
	Surface Infrastructure (WBS 3000)	1,994	1,694	1,494	Fan Facilities (Fresh/Return)
	Total	**4,998**	**4,191**	**3,346**	
	Transformer Capacity			5,000	
				67.0%	

16.18.2 Compressed Air

The existing compressed air supply to the mine is provided by two AWA-300 (225 kW) units each rated at 2,515 m^3/h at 690 kPa and one TS325-450H (335 kW) unit rated at 3,740 m^3/h at 860 kPa. All 3 compressors are rented an equipment supplier. The two AWA 300 units are located at the portal to the main ramp and the TS325-450H unit is located at RAR #1, a surface distance of about 300 m apart. The AWA300 compressors feed into a 10 cm pipeline in the ramp and the single TS325-450H unit feeds into a 15 cm line in RAR #1. There is a spare 15 cm line currently located within the raise that is not currently in use. The current compressors supply up to 8,770 m^3/h of compressed air to the underground workings.

The estimated compressed air requirements for the LOM have been calculated to support for the long hole drills, air leg drills (stopers, jacklegs), and air pumps amount to 9,150 m^3/h, these requirements are summarized in Table 16-16. The jumbos have their own 'on-board' air supply. A 20% allowance has been applied on top of this air requirement for miscellaneous use and leakage for a total consumption of 10,980 m^3/h. It is assumed that no surface facilities are being fed compressed air from the compressors installed at the top of the RAR (Stantec, 2021).

Table 16-16: Summary of Underground Compressed Air Usage

Pneumatic Equipment	# of Units	Compressed Air Consumption per Unit (m^3/h)	Fleet Air Consumption Rate (m^3/h)
Stope Mate LH Drill	7	841	5,890
Air Leg Drills	12	255	3,060
Air Pumps	6	34	200
Compressed Air Required for Primary Consumers			**9,150**
Miscellaneous Usage Allowance (20%)			1,830
Total Compressed Air Required			**10,980**

Note: Totals may not sum due to rounding.

To provide the recommended volume of air it is anticipated that Harte Gold will require two additional operating 335 kW compressors located at the RAR #1 location. As there may be maintenance events or other service interruptions in the operations of one of the compressors it may be preferrable to add a third 335 kW compressor as a stand-by unit. The actual air requirements and anticipated impact to production will determine whether two or three additional compressor units are added to the system. These compressors are forecast to be rented from an equipment supplier similar to the current TS325-450H. An air receiver will be installed at RAR #1 to allow for moisture to be removed from the air.

The additional compressors will be located at the RAR #1 location and will feed the spare 15 cm pipeline in the RAR #1. Upon installation of these units the two AWA-300 diesel units located at the portal will be decommissioned and returned to the equipment supplier.

The underground distribution system will make use of the two existing 15 cm diameter lines from surface to the 200 m level. At the 200 m level these lines will be connected into a 15 cm diameter schedule 40 header, which will direct the air towards the north and south ramps. Both the north and south ramps will include a 15 cm schedule 40 line distribute the air to the bottom of the ramp. A 15 cm schedule 40 line will be required to feed air from the 200 m level up ramp to the middle zone and to the bottom of the middle ramp. The existing 10 cm compressed air line that runs from the portal will be connected to the 200 m level 15 cm header line to allow for feeding compressed air to the upper areas of the workings. All the compressed air piping will be carbon steel, grooved pipe, and include Victaulic couplings. The line sizes have been selected to allow for a 620 kPa minimum pressure at the bottom of each ramp.

16.18.3 Dewatering and Process Water Supply

The water entering the mine is the sum of seepage from the surface via faults, fissures, boreholes, and water from the surface reservoir or the pond, which is used as “make-up” for the water balance underground. The amount of water used for human needs is insignificant, and the water contained in the paste backfill will be consumed by hydrating the binder. The absolute amount of water used by the mining equipment is not relevant, as it is recycled within the underground system.

The historical high and low mine dewatering monthly average and ranges are as follows:

- High April average 477 m^3/d (range 174-815 m^3/d)
- Low January average 134 m^3/d (range 0-626 m^3/d).

Figure 16-32 shows a process flow diagram of the current and future water settling, pumping and supply strategy (Stantec, 2021).

A cascading pump and settling system is used to remove excess water from the mine.

Due to the fact that process water is provided in large part from the underground sumps, it is critical that these sumps have suitable settling capacity to provide high-quality water for use in the drilling equipment.

There are significant seasonal fluctuations in the amount of groundwater entering the mine and during low inflow periods it will be necessary to supply clean water from the surface pond to supplement the clean water recovered from the underground sumps.

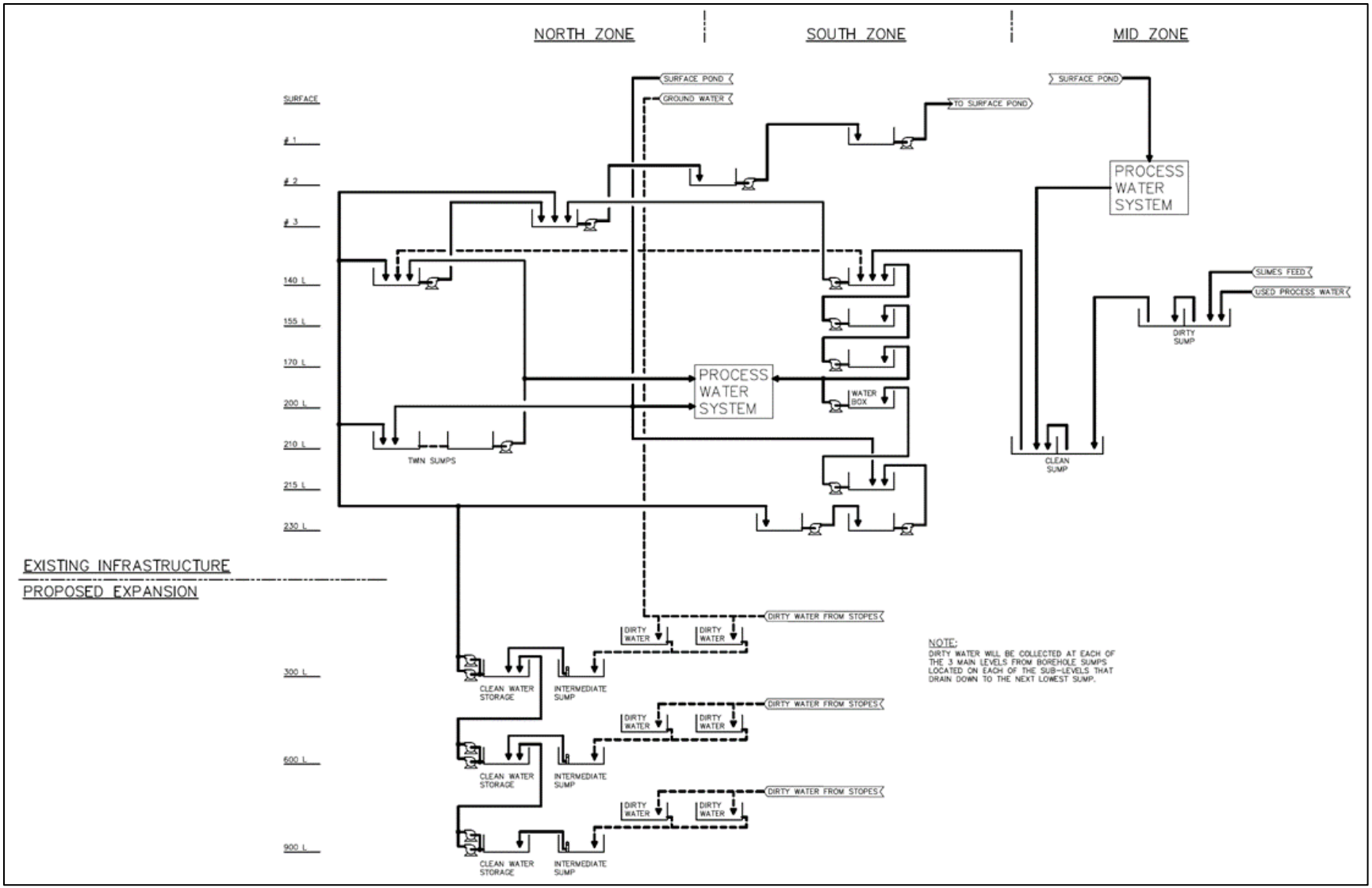


Figure 16-32: Current and Future Underground Water Process Flow Diagram

16.18.4 Communications

Underground communications are through a Becker Varis VHF leaky feeder network that is advanced with development and provides voice and data communication between underground workers in different work areas and with surface personnel.

16.18.5 Central Blasting

Central blast line is advanced with development and the main blast box is on surface at the shift supervisor's station.

16.18.6 Refuge Chambers and Escapeways

All producing areas of the underground workers have a secondary route of egress and there are several underground refuge chambers, both temporary and permanent. Figure 16-33 provides a detailed visual of the current refuge chamber and escapeway system at the Sugar Zone Mine.

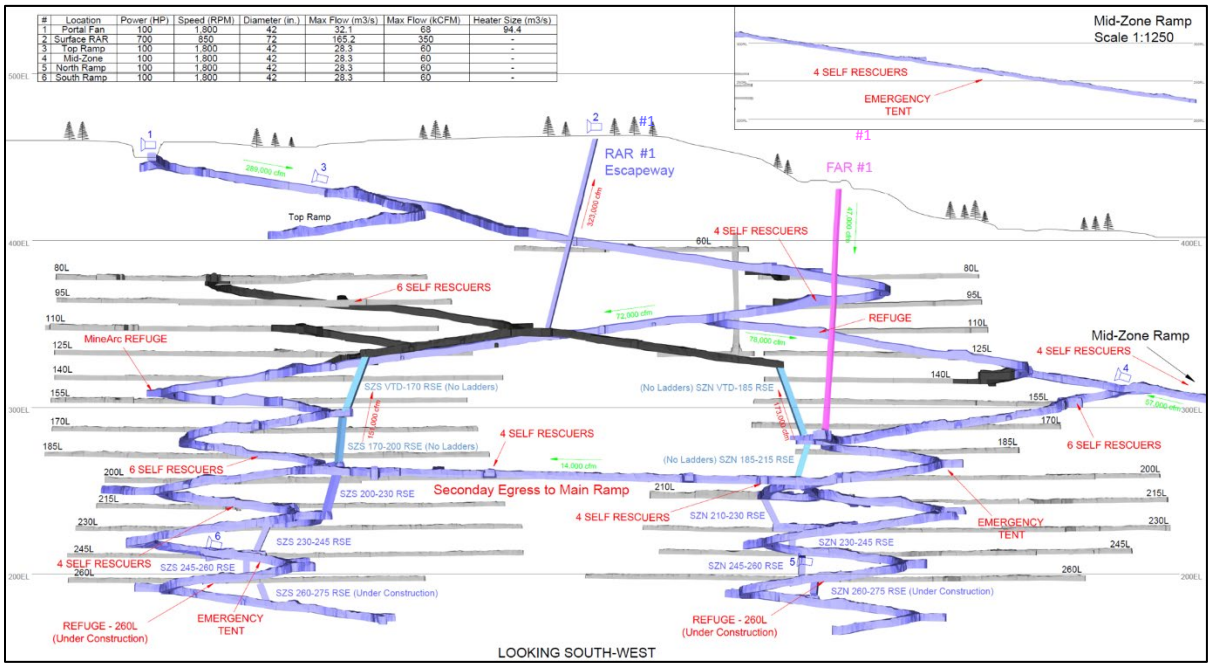


Figure 16-33: Current Mine Escapeway Longsection (December 2020)

The secondary egress route in the North and South Zones are extend to each sublevel via the ventilation raises. The 4 x 4 m ventilation raises approximately 15 m long and are drilled and blasted with longhole drilling equipment, these raises are then supported and outfitted with timber ladderways. This system of ladderways continues up to the 100 Level RAR drift and the final egress to surface is through the RAR #1 emergency egress ladderway.

The secondary egress route in the Middle Zone will follow the same design principles as in the North and South Zones and will be extended to each sublevel via the ventilation raises as described above. The primary difference is that the final connection between the underground workings and surface will be via the Middle Zone RAR #2 which is expected to be completed in 2021.

As mine development continues towards the Middle Zone and further down each ramp refuge chambers are planned to be constructed every 60 m in vertical depth. As development advances rapidly there may be temporary emergency tents installed closer to the active working faces to ensure adequate refuge locations as the permanent ones are constructed. Harte Gold also owns one skid mounted portable refuge station that can be located closer to the working face as development drifts extend further from main infrastructure.

16.19 Mine Surface Facilities

This section provides further detail on the site infrastructure that services the underground mine and mobile equipment. Some of the facilities described may also service the process plant or other surface operations. Figure 16-34 provides an overview of the mine site area.

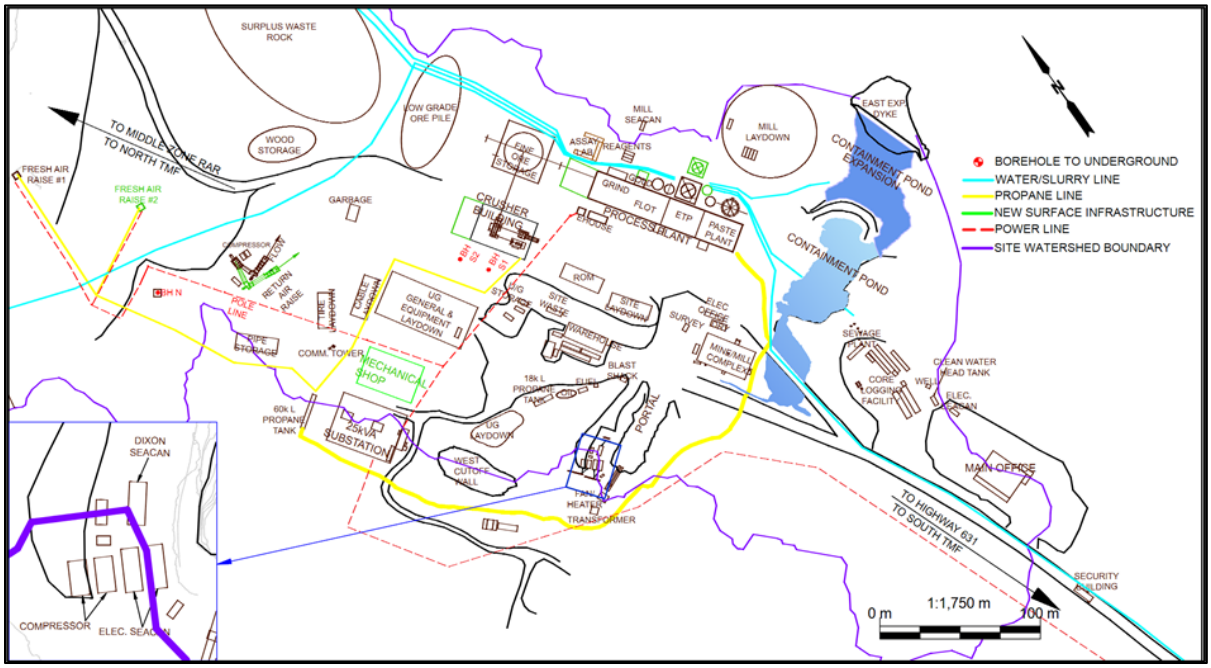


Figure 16-34: Sugar Zone Mine Site with Expected Future Infrastructure

16.19.1 Workshop

The current maintenance workshop is a Quonset-style structure on a concrete pad, with fabric walls and roof. It is approximately 8 m wide x 20 m long. This facility allows for routine maintenance and

some larger jobs to be completed, but will become limiting as the mine equipment fleet grows and production rate increases.

In 2022, a maintenance workshop will be built on surface; it is envisioned to be approximately 24 x 24 m and will provide two drive-through repair bays capable of accommodating the largest mechanical equipment (30-tonne trucks, 2-boom jumbos). The workshop will complement the planned underground workshops and provide a clean working environment for larger maintenance tasks that are difficult or impractical to complete underground. The workshop is expected to have:

- A large, enclosed wash bay to ensure that equipment is properly cleaned to facilitate maintenance work
- Overhead crane for major component repair
- Dedicated area for repair of pneumatic drilling and pumping equipment
- Toolbox storage area and small tool-crib
- Full heating, lighting, and appropriate electrical supply.

16.19.2 Ventilation Surface Infrastructure

16.19.2.1 Portal Ventilation

Currently there are two 110 cm diameter 100 HP fans located on top of the portal with steel ducting conducting the air into the portal. The air is pulled through two propane fired burners which heat the air in winter.

16.19.2.2 Sugar Zone Return Air Raise

The RAR #1 is located north of the crusher building. The raise is equipped with an emergency egress ladderway and the ducting on top of the collar includes an airlock personnel door. The return air raise currently has a 522-kW fan which pulls vitiated air out of the mine workings, this fan is connected to the raise collar by fabricated steel ducting supported by poured concrete footing.

In Q3 2024 an additional 522 kW fan will be installed along with the required ducting. The poured concrete footings for the fan and ducting are already in place.

At the RAR #1 location there is an electrical room in a shipping container for fan power supply and controls.

There is also a small semi-permanent building which houses a 335-kW air compressor which feeds a 15 cm air pipeline down the RAR.

16.19.2.3 Fresh Air Raise #1

In 2019 a 2.8 x 2.8 m alimak raise was completed, this FAR #1 has a collar on surface that is north of RAR #1. In Q4 2020 Harte Gold constructed a concrete collar around this raise and created a compacted aggregate pad for installation of a 56-kW fan and propane fired burner for the supply and heating of additional air to the underground workings. The fan and heater are connected to the site power supply and to the propane tank through appropriately protected cable and pipeline conduits.

16.19.2.4 Middle Zone Return Air Raise

In 2021 the Company will be developing an additional alimak raise from the underground workings to surface. This RAR #2 will be 4 x 4 m in cross sectional area and connect the Middle Zone mine workings to surface at a location north of the North TMF. This raise will have a poured concrete collar and elbow duct with airlock personnel door and flume on surface. The fan that pulls air up this raise will be located underground and installed in a bulkhead. The raise will be outfitted with an emergency egress ladderway and there will be a service road between the main mine site complex and the RAR #2 location.

16.19.2.5 Fresh Air Raise #2

In 2022 the Company will develop an additional and final ventilation raise connection between surface and the North Zone. The FAR #2 is anticipated to have a collar location between RAR #1 and FAR #1 and will be excavated with raise bore technology to create a 4.3 m diameter circular cross section hole. Detailed design work to determine the specific underground and surface collar locations will be undertaken in 2021. FAR #2 will have a 335-kW fan and a propane fired burner installed on a poured concrete footings and connected to the raise via steel ducting anchored to a poured concrete collar.

16.19.3 Explosives Magazines

The Sugar Zone Mine is located in a well-serviced mining district with easy access to provincial highways, which allows for regular, uninterrupted explosives deliveries. Upon delivery, explosives are offloaded and immediately transferred underground to explosives magazines.

16.19.4 Offices and Change House

Currently the mine operations complex consists of several ATCO trailers that provide office space for mine operations, technical services, maintenance, and administration personnel. The mine operations complex also has space allocated for pre-shift safety meetings and the mine rescue team. Mine change houses are also found in this complex; they provide clean facilities for workers to shower and change at the beginning and end of their shift.

16.19.5 Warehouse

Currently the warehouse facilities on site are shared between mine and mill operations, and are made up of several shipping containers and lay-down areas that allow for the receiving, storage, and distribution of consumables and larger supplies as required by the Operation.

In 2022, upon completion of the new workshop building referenced in Section 16.19.1 the current workshop building will be converted into the primary on-site warehouse facility.

The on-site facilities are supplemented by a steel-clad Quonset-style building in White River; this warehouse can receive and store materials and supplies that are not immediately needed on-site.

16.19.6 Fuel and Oil Storage

All diesel is purchased from a commercial vendor and transported to site by a fuel tanker truck. Diesel is stored on surface in a 25,000-L tank which is used to fuel all underground and surface equipment as well as any diesel generators or compressors.

No gasoline is stored on site, all light vehicles are fuelled at a commercial gas station in White River.

Oil and other lubricants are transported to site in 20-L pails or in 205-L drums. The pails are stored in a designated shipping container and the drums are stored on a designated concrete pad.

17 RECOVERY METHODS

The existing Harte Gold Sugar Zone process plant currently produces a gold doré bar and a bagged gold concentrate through gravity concentration and flotation circuits, respectively. The objective of this Feasibility Study is to understand the impact of expanding the plant's throughput to 1,200 t/d and either continuing to produce a combination of gold doré bar and a minimum 100 g/t Au-grade flotation concentrate, or to produce only gold doré bar through on-site leaching.

The current performance and possible expansion options were modelled following a review of existing comminution circuit performance, using the most recent operating data from August 2020. The model revealed that bottlenecks for the 1,200 t/d throughput expansion will be the HP100 secondary crusher and the existing 800 kW ball mill.

Based on current operational performance, to achieve 1,200 t/d an HP200 is required to replace the existing secondary crusher (HP100), and a second ball mill is required to process ore up to a design BWi of 16.1 kWh/t at a 75 µm to 100 µm grind size. The circuit will require a second cyclone online to maintain the grind target.

The option to replace the secondary crusher with a SAG mill was considered but subsequently ruled out due to previous testwork and high required capital costs. Using a rod mill was also eliminated as an option early in the study, due to operational complexities and more favourable alternative options.

Three grinding circuit configurations were studied: a new ball mill in parallel; a new ball mill in series; and a combination of parallel and series. Based on metallurgical performance, construction complexity and capital cost considerations, the new ball mill in series was selected as the preferred option for the 1,200 t/d expansion.

To determine the best combination of gold doré bar and flotation concentrate to be produced for the 1,200 t/d expansion, three options were studied: increasing capacity to the existing gravity/flotation concentrate circuit; constructing a continuous carbon-in-leach (CIL) circuit; and a batch leaching followed by a Merrill-Crowe (MC) circuit. After evaluating the options, it was decided to increase capacity to the existing gravity/flotation concentrate circuit with the addition of a new cleaner flotation stage, producing a combination of gold doré bar and a 100 to 150 g/t Au flotation concentrate.

In addition to the comminution changes mentioned above, this design requires: upgrading the existing gravity circuit; increasing gold room capacity; adding two new rougher flotation tank cells; adding a new single-column flotation cell; and adding a new tailings thickener.

17.1 Process Description and Process Block Diagram

The process plant stages for the existing and future expanded plant are as follows:

- Primary jaw crushing and secondary cone crushing with intermediate screening
- Fine ore feed storage (stockpile)
- Grinding circuit with classification tower
- Gravity concentration circuit
- Gold room with furnace
- Flotation circuit (addition of a cleaner flotation cell for expanded circuit)
- Concentrate thickening and filtration with transportation loadout
- Tailings thickening circuit
- Reagents area
- Associated services such as water and air.

The 1,200 t/d expansion process block diagram is shown in Figure 17-1.

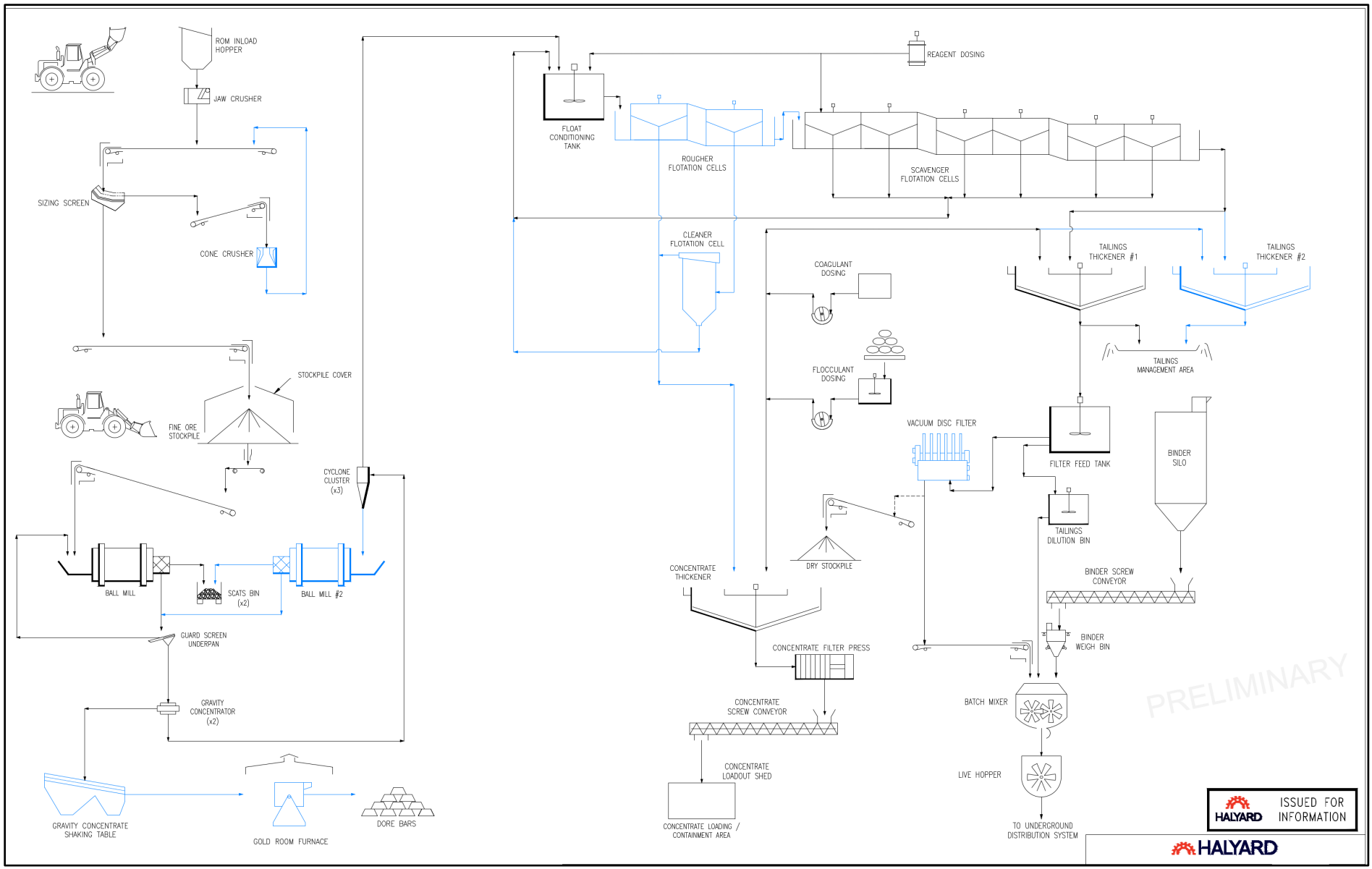


Figure 17-1: 1,200 t/d Expansion Process Block Diagram with Additional or Upgraded Equipment Highlighted in Blue

17.2 Major Process Design Criteria

The existing and proposed expanded process plant were designed according to the major design criteria given in Table 17-1 for throughput (tonnage) and design data, as well as availability and hours of operation.

Table 17-1: Major Process Design Criteria

Criteria		Unit	Design Data		Source[1]
			800 t/d	1,200 t/d	
Process Plant General	Annual Throughput	dmt/a	288,800	433,200	4
	Daily Throughput	dmt/d	800	1,200	1
	Operating Days/Year	d	361	361	1
ROM Physical Characteristics	Moisture	%	5.0	5.0	1
	Solid Particle Density (Ore)	t/m^3	2.75	2.78	1
	Abrasion Index (Bond Ai)	g	0.616	0.616	1
	Crushing Work Index (CWi)	kWh/t	12.7	12.7	1
	Ball Mill Wi Bond–106 mesh	kWh/t	16.1	16.1	1
ROM Chemical Characteristics	Au Head Grade	g/t	10	7.50	2
	Ag Head Grade	g/t		1.50	2
	S_{total}	%	1.10	1.10	2
Crushing Circuit	Circuit Configuration		Closed circuit–primary jaw crusher & secondary cone crusher	Closed circuit–primary jaw crusher & secondary cone crusher	
	Hours per Shift	h	12	10.5	1
	Shifts per Day	#	2	2	1
	Days per Week	#	7	7	1
	Operating Days per Annum	d	357	357	2
	Availability	%	75	75	2
	Operating Hours per Annum	h/a	6,426	5,633	4
	Design Hourly Feed Rate	dmt/h	45	120	4
	Feed Size F_{100}	mm	450	450	1
	Feed Size F_{80}	mm	158	220	1
	Product Size P_{100}	mm	12	16	4
	Product Size P_{80}	mm	8.5	11	4
Grinding & Gravity Concentration Circuit	Circuit Configuration		Closed circuit–single ball mill	Closed circuit–two ball mills in series	
	Hours per Shift	h	12	12	1
	Shifts per Day	#	2	2	1
	Days per Week	d	7	7	1
	Days per annum	d	361	361	2
	Availability	%	92	92	1
	Operating Hours per Annum	h	7,971	7,971	4
	Hourly Feed Rate	dmt/h	36.2	54.3	4
	Daily Feed Rate	dmt/d	800	1,200	1
	Product Grind Size (P_{80})	µm	75	75	1
	Hydrocyclones Type		HC420-20-0/A	HC420-20-0/A	5
	Number of Hydrocyclones	#	1 op, 1 sb	2 op, 1 sb	
	Gravity Concentrators Type		Falcon SB 1350	Falcon SB 1350	
	No. of Gravity Concentrators	#	2	2	
	Gravity Concentrators Au Recovery	%	75	62.5	2

Criteria		Unit	Design Data		Source[1]
			800 t/d	1,200 t/d	
Flotation Circuit	Rougher Residence Time	min	10.7	13.1	4
	Rougher No. of Cells	#	2	2	2
	Scavenger Residence Time	min	22.3	20	4
	Scavenger No. of Cells	#	4	6	2
	Cleaner Residence Time	min	NA	32.8	4
	Cleaner No. of Cells	#	NA	1	2
Concentrate Dewatering	Concentrate Thickener Type		High rate	High rate	
	No. Concentrate Thickeners	#	1	1	
	Concentrate Thickener Unit Area		3.2	1.8	4
	Cake Moisture	%	9	9	
Tailings Dewatering	Tailings Thickener Type		High compression	High compression	
	No. Tailings Thickeners		1	2	
	Tailings Thickener #1 Unit Area	t/d/m^2	7.3	7.3	4
	Tailings Thickener #2 Unit Area	t/d/m^2	NA	6.9	4
Reagents	Flocculant Type		Magnafloc 10	Solenis Zetag 4105	1
	Flocculant Consumption	g/t	25	10	1
	Coagulant Type		NA	Solenis Amerfloc 485	1
	Coagulant Consumption	g/t	NA	25	1
	Collector Type		PAX	KAX-51-33% Strength	1
	Collector Consumption	g/t	50	400	1
	Promoter Type		Cytec A407	Solvay A407	1
	Promoter Consumption	g/t	25	40	1
	Frother Type		MIBC	MIBC	1
	Frother Consumption	g/t	20	15	1

Note: [1] Sources: 1 = Owner; 2 = engineer; 3 = testwork; 4 = derived from other data; 5 = vendor.

17.3 Process Plant Layout

Figure 17-2 shows the new equipment to be installed during expansion (in blue/bold), and its arrangement in relation to the existing process plant equipment.

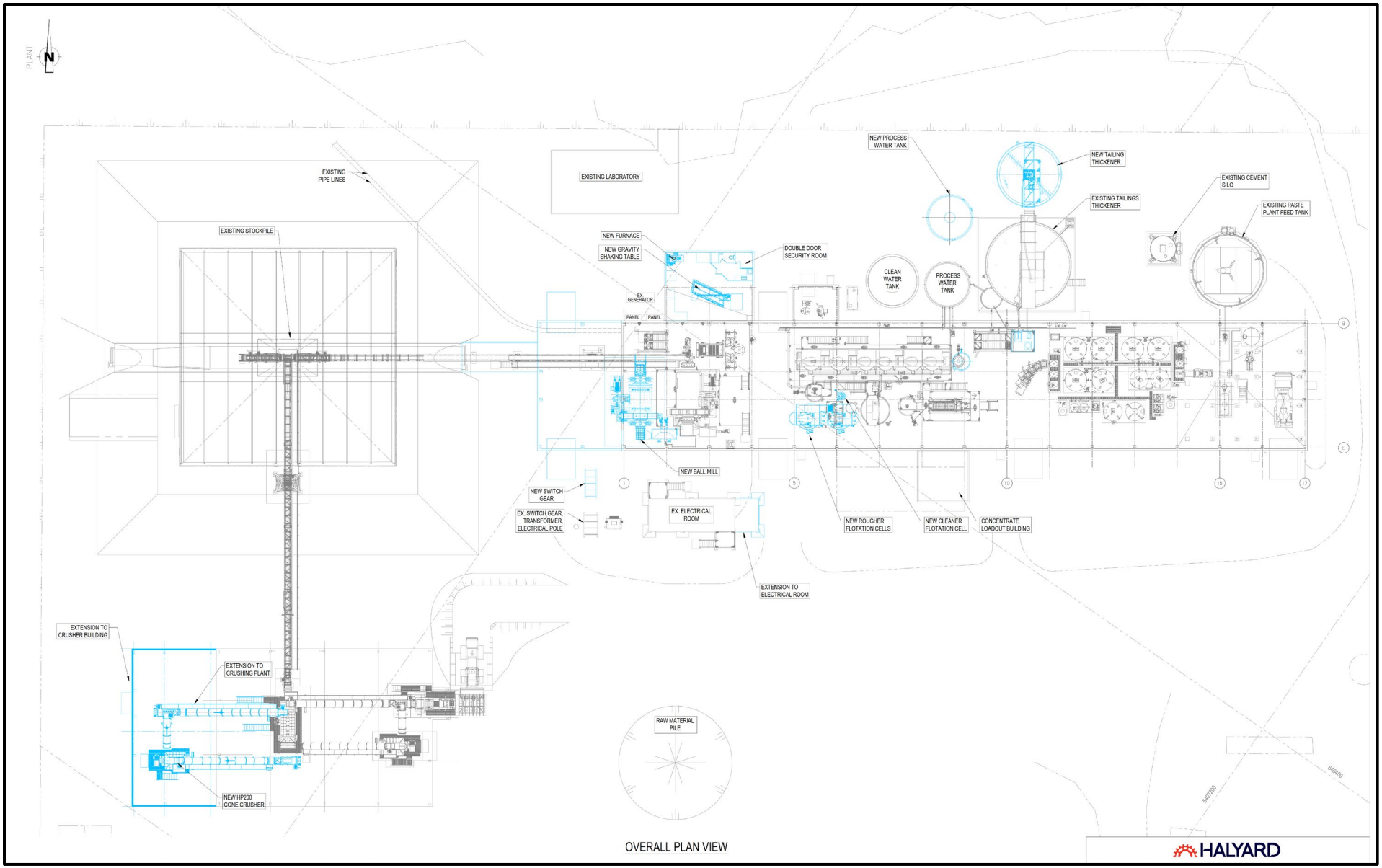


Figure 17-2: 1,200 t/d Expansion Process Plant Layout with Additional or Upgraded Equipment Highlighted in Blue

17.3.1 Process Plant Modifications

17.3.1.1 General

This study examined the impact of additional throughput on the processes and equipment, and concluded that the following changes were required:

- Extend the north end of the pre-engineered crusher building and additional larger secondary cone crusher to increase capacity of the crushing circuit
- Minor upgrades to materials handling systems (conveyors) by changing selected motors
- Minor changes to the mill feed conveyor feeders by changing to sliding feed beds
- Add a 550-kW ball mill in series with existing 800 kW ball mill in a closed circuit
- Extend the north end of the pre-engineered mill building to enclose the expanded ball mill circuit
- Add a replacement cyclone cartridge for an additional online cyclone and spare standby cyclone for the increased capacity
- Expand gold room with higher capacity equipment
- Add two additional rougher flotation cells and one new cleaner flotation cell
- Add a new electrical room, complete with motor control centre and transformer
- Add a tailings thickener with additional tailings pumps and pipeline to the North TMF
- Add selected additional reagents storage and pumping equipment
- Add ancillary services equipment such as a new process-water tank, air compressors, HVAC system, and gold room.

The following sections examine the individual process plant areas, with a view to upgrading existing equipment to meet the 1,200 t/d throughput requirement.

17.3.2 Crushing Circuit

The crushing area consists of three modules, a jaw/primary crusher unit, a sizing screener unit, and a cone/secondary crusher unit. The ROM material from underground is fed directly to the primary jaw crusher (through a grizzly) and ROM bin. This material is then passed through a vibrating screen with a 13 mm aperture; the undersize reports to the crushed fine ore feed stockpile, and the oversize to the secondary crusher.

The jaw crusher unit includes a ROM bin, vibrating grizzly feeder, and Metso C80 jaw crusher to reduce the material from a 400 mm top size to a P_{80} of 45 mm. The jaw crusher is fed by a vibrating grizzly feeder set to scalp at 5 cm (2); its oversize reports to the jaw and the undersize −5 cm (−2") bypasses directly to the screen feed conveyor. The screen feed conveyor is equipped with a self-cleaning crossbelt magnet and a metal detector that will automatically stop the belt when triggered. There is

also a manual cleaning magnet on the existing HP100 cone crusher feed conveyor. Tramp metal will then be manually removed from this magnet.

The Metso double-deck vibrating screen is a model ES202 (1.87 x 4.9 m) with a 38 mm top deck and a 15 to 16 mm bottom deck.

The current cone/secondary crusher circuit consists of an intermediate surge bin, vibrating feeder, and HP100 cone crusher. The material from this cone crusher is returned to the jaw crusher discharge belt to return to the classification screen.

The secondary crushing circuit configuration requires an upgrade of the secondary cone crusher from an HP100 to an HP200 or equivalent to sustain 1,200 t/d. In addition to the existing three modules that comprise the current crushing circuit (C80 jaw crusher, HP100 cone crusher, and ES202 sizing screen), a fourth module will be added for the new HP200, with modification at the oversize screen chute capable of feeding the HP200 module or diverting the feed to the existing HP100 module only when the HP200 is down for maintenance to maintain crushing capacity, albeit lower than the required new expansion throughput.

This new circuit model was set up using Bruno software, including the new HP200 cone crusher with a standard medium set and a 16 mm closed side set (CSS) opening. The existing ES202 screen was modelled, as well as the existing C80 jaw crusher, which was simulated at 55 to 85 mm jaw gap opening.

The Bruno simulation was performed at a nominal capacity of 76.9 t/h and predicts a secondary crusher load of 48%, shown Figure 17-3. However, a higher loading of 85% is recommended, to “choke feed” the cone crusher; such loading is predicted at 120 t/h fresh feed, as shown in Figure 17-4. Choke feeding is beneficial as it provides even wear and efficient crushing. At low loads, the rock may “fall through” the gap and build up an unwanted coarse circulating load, leading to inefficient operation. For this reason, it is inadvisable to run both the HP200 and HP100 at the same time. The option of replacing the HP100 with an HP200 unit would have required significant sizing and changes to the bin and structure, thus a separate module was preferred from a capital cost and operational interruption point of view.

The C80 jaw crusher closed side setting of 55 mm will serve the nominal feed rate; however, it is recommended to open it to 75 to 85 mm to choke feed the secondary cone crusher. The secondary cone crusher will work in a closed circuit with the ES202 vibrating screen (1.87 m x 4.9 m); at 120 t/h the screen shows low loading and will not become a constraint. Thus, a screen upgrade is not required. However, some minor motor upgrades will be required to the materials-handling conveyors.

A summary of the equipment required for the crushing and stockpile areas’ expansion from 800 t/d to 1,200 t/d is given in Table 17-2.

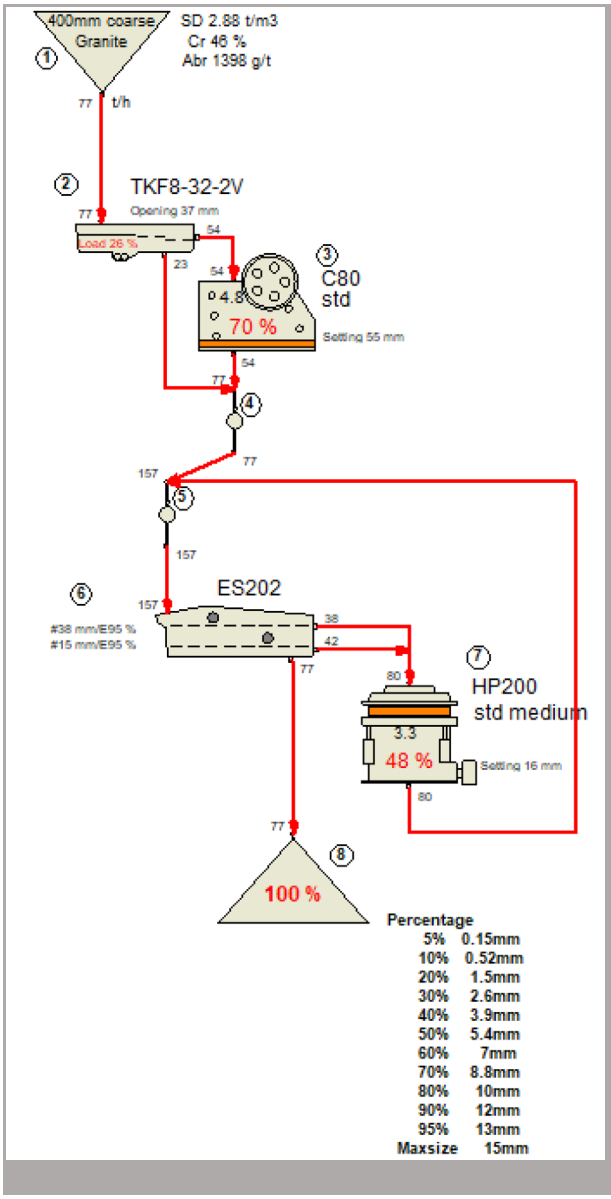


Figure 17-3: Crusher Circuit Nominal Flow Diagram 77 t/d

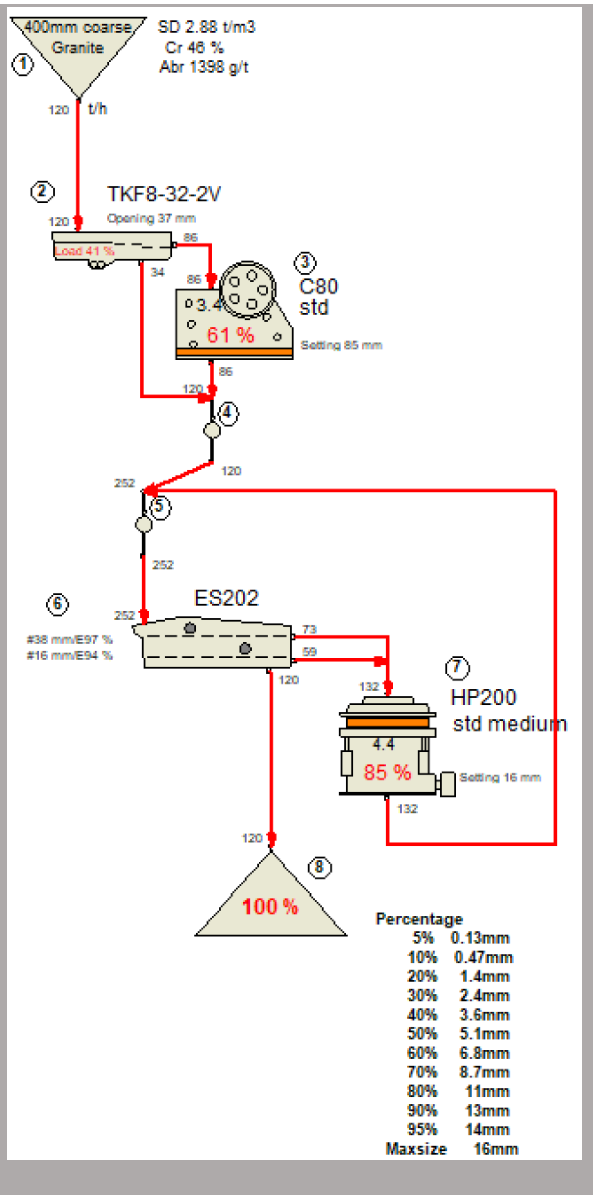


Figure 17-4: Crusher Circuit Design Flow Diagram 120 t/d

Table 17-2: Crushing and Stockpile Areas Major Equipment for 800 t/d and 1,200 t/d

	Equipment	Unit	800 t/d	1,200 t/d
Grizzly Feeder	Type		TKF8-32-2V	TKF8-32-2V
	Width	m	0.7	0.7
	Length	m	3.3	3.3
	Openings	mm	50	50
Primary Jaw Crusher	Type		C80 Quarry	C80 Quarry
	Model		Metso C80	Metso C80
	CSS	mm	45	55
Classification Screen	Type		Horizontal Double Deck	Horizontal Double Deck
	Model		ES202	ES202
	Width	m	1.9	1.9
	Length	m	4.9	4.9
	Top Deck Slot Width & Length	mm	38	38
	Lower Deck Slot Width & Length	mm	13	16
Secondary Crusher	Type		Cone Crusher	Cone Crusher
	Model		HP100	HP200
	CSS	mm	15	16
Fine Ore Stockpile	Feeder Capacity Peak	dmt/h	50	60
	Feeder Type		Pan Feeder	Slider Bed Feeder

17.3.3 Fine Ore Feed Stockpile

Some minor motor upgrades will be required to the materials handling conveyors. The two mill feed conveyor feeders within the tunnel under the fine ore feed stockpile will also need to be upgraded to slider-bed types.

17.3.4 Grinding and Gravity Concentration Circuit

The existing mill feed conveyor is fitted with a weightometer to monitor and control the feed rate to the ball mill. The weightometer is also used to proportionally add water to the milling circuit based on feed tonnage.

The grinding area is currently made up of a single 3.05 x 5.79 m (10 ft diameter x 19 ft long), 800 kW ball mill in closed circuit with a cyclone pack. The mill is designed to produce a P_{80} 75 µm grind size.

Ore is fed to the existing ball mill at a controlled rate, and water is added to the feed chute to achieve the desired milling density. Product from the ball mill discharges through a trommel with the oversize reporting to the scats bunker, where it is periodically removed by the skid steer loader and returned to the circuit.

Trommel undersize gravitates to the guard screen feed hopper where it is further diluted to achieve the required cyclone feed density before entering the pair of gravity concentrators. The gravity concentrators operate so that their backwashing and flushing cycles are staggered.

Once the material has passed through the gravity concentrators, the cyclone feed pump delivers slurry to the cyclone cluster, which currently has 2-cyclone carousel arrangement (a duty and a standby). The cyclone underflow gravitates back to the ball mill, while the cyclone overflow gravitates to the sulphide flotation circuit.

Two vertical spindle sump pumps service the mill area, one located at the feed end of the mill, the other at the discharge end. The concrete floor under the mill area is sloped to the sumps to facilitate clean-up. Grinding media for the mill is introduced by a dedicated kibble and the overhead gantry crane.

A simulation revealed that the existing 800 kW ball mill could match the predicted crushing circuit capacity of 918 t/d only with a BWi of less than 13.5 kWh/t, but would need significantly softer ore to operate at 1,200 t/d.

A new ball mill (BM2) operating with the existing ball (BM1) mill will provide the expansion capacity. A ball mill was chosen given that the site is familiar with operating the existing mill, and as mentioned above, previous testwork and financial considerations showed that there would be no advantage in adding a SAG or rod mill.

Three options for the second ball mill were considered—series, parallel, or a combination that would allow for series flow with the ability to transfer the ore feed to the second ball mill when the first ball mill is undergoing maintenance.

The two ball-mill series configuration and mass balance are shown in Figure 17-5. The ore feed will be delivered to the BM1 feed chute with process water to achieve the target discharge percent solids. BM1 will work in an open circuit, and discharge into the existing gravity-feed pump box together with BM2 discharge slurry. The gravity circuit and the classification circuit routing will be the same as the present circuit; however, the final sizing of pumps and lines will be confirmed in detailed engineering. The cyclone underflow will be directed to BM2's feed chute. BM2 discharge will ultimately report to the gravity-feed pump box into which BM1 discharges. BM2 operates in a closed circuit with the cyclones, with the cyclone overflow reporting to flotation at the desired grind target.

The parallel option would require significant and costly changes to the feed conveyor, whereas the series option could use the existing ball mill feed system and conveyor. The viability of this parallel option did not warrant further effort in this study.

The combination (or hybrid) option would require a temporary feed diversion to be installed to the second ball mill, also requiring significant changes to the current ball mill feed system. The viability of this combination option did not warrant further effort in this study.

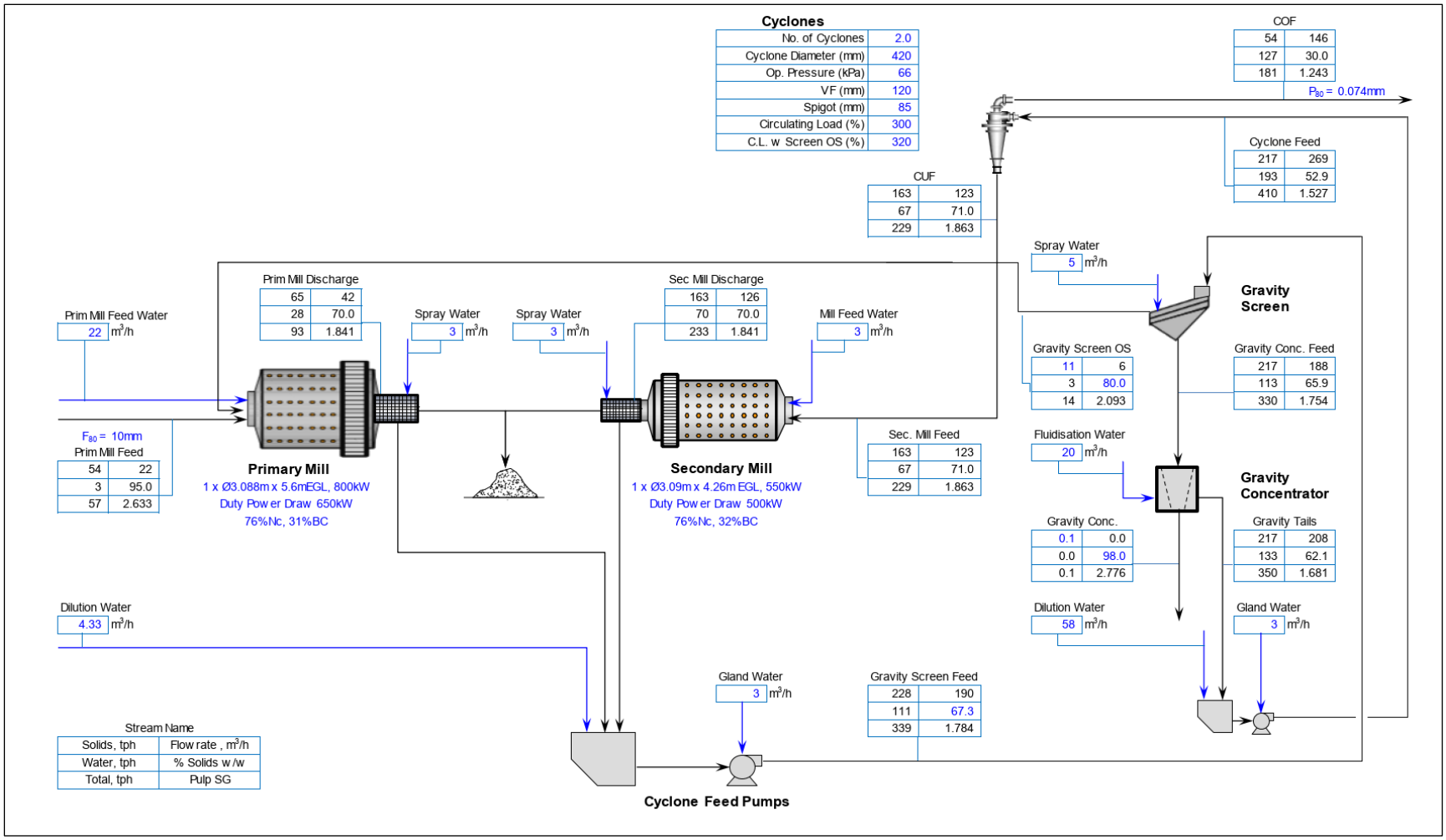


Figure 17-5: Flowsheet and Mass Balance in Series Ball Mills

A 3.09 m diameter x 4.60 m EGL ball mill with a 550-kW variable-speed drive (VSD) was chosen as the second ball mill and will operate in series downstream of the existing 3.09 m diameter x 5.6 m EGL ball mill with 800 kW VSD.

The selection of a VSD for BM2 will help optimize power draw in case of feed-blend hardness changes. It is recommended that the mill be operated at a higher-than-26% ball charge, and the mill be slowed down to cater to the required power. In this new configuration, milling efficiency improvement can be expected when selecting a 75 mm ball for BM1 and a 50 mm ball for BM2.

A simulation done for the current operation data of the existing mill showed that the existing 800 kW drive on BM1 is "oversized," as a 46% ball charge would be required to draw its maximum power at 80% of critical speed. This can be explained in part due to the lower, 9 to 10 kWh/t BWi ore currently being treated at the plant, compared to the 16.1 kWh/t that was used for the initial process plant design. The geometry of the mill, according to modelling for BM1, may require a ball retainer if its ball charge exceeds 34%. Care should be taken if a higher ball charge is attempted; the structural ball charge maximum stated by the mill manufacturer should be reviewed.

The mill sizing relies on the BWi of 16.1 kWh/t (106 µm screen) and a grind target of 75 µm, as per the original design. Although recent testwork samples of ball mill feed show a significantly lower BWi of 9.9 kWh/t (106 µm screen) to 11.6 kWh/t (150 µm screen) for the Sugar Zone, future ore zones including the proximal Middle Zone and deeper reaches of the North and South Zones have yet to undergo robust comminution testwork, hence a higher design BWi of 16.1 kWh/t provides a conservative basis for the design.

The ball mill discharges to the gravity circuit and is conveyed by pump to a guard screen that protects the gravity concentrators from >3 mm particles. The underflow from this screen subsequently goes to a pair of Falcon SB1350 gravity concentrators. The cycle times of the gravity concentrators is staggered so that they do not flush simultaneously and overload the hopper in the gold room.

The gravity concentrators were previously sized to recover 75% of the gold with a grade of 1,875 g/t Au. This is dependent on the quantity of free gold reporting to the mill. The existing pair of Falcon SB 1350 gravity concentrators and existing guard screen have capacity to accept the expanded throughput with a drop in the recovery percentage expected accordingly.

The expansion requires an additional cyclone, for a total of three cyclones, with two online and one on standby. A new cyclone pack consisting of a 4-cyclone carousel will replace the existing double configuration as this is the next-highest-capacity configuration available.

A summary of the equipment required for the grinding and gravity concentration circuits expansion from 800 t/d to 1,200 t/d is shown in Table 17-3.

The capacity of the existing grinding and gravity concentration circuit pump boxes, pumps, and associated piping needs to be upgraded. The design considers a single new discharge box for both ball mills, with a duty and standby gravity concentrator circuit pump and a new cyclone feed pump box, with duty and standby cyclone feed pumps. A new sump pump will be installed in the sloped floor at BM2.

Table 17-3: Grinding and Gravity Concentration Circuit Major Equipment for 800 t/d and 1,200 t/d

Equipment		Unit	800 t/d	1,200 t/d
Mill Feed Conveyor (Existing)	Tonnage	dmt/h	39.9	59.8
Ball Mill (Existing)	Mill Diameter	m	3.1	3.1
	Mill EGL	m	5.6	5.6
	Applied Power at Pinion	kWh/t	17.1	12.2
	Pinion Power Consumption	kW	683	664
	Selected Motor Size	kW	800	800
	Mill Speed	%Critical speed	77	76
		rpm	24.6	18.7
	Media Volume	%	32–35	31–37
	Mill Discharge % Solids	%	70	70
Ball Mill (New)	Mill Diameter	m	NA	3.0
	Mill EGL	m	NA	4.6
	Applied Power at Pinion	kWh/t	NA	9.3
	Pinion Power Consumption	kW	NA	508
	Selected Motor Size	kW	NA	550
	Mill Speed	%Critical speed	NA	76
		rpm	NA	18.7
	Media Volume	%	NA	32–37
	Mill Discharge % Solids	%	NA	70
Hydrocyclones	Type		2 x Multotec HC420-30-0, Spigot 170-HS	3 x Multotec HC420-20-0/A-B/90, Spigot 120-HS
	Operating Pressure	kPa	60	40
Classification Screen	Type		Horizontal Deck, slotted Screens	Horizontal Deck, slotted Screens
	Width	m	1.5	1.5
	Length	m	3.0	3.0
	Angle	°	5.0	5.0
	Slot Width and Length	mm	2 x 6	2 x 2
Gravity Concentrator	Type		Falcon SB 1350	Falcon SB 1350
	# of Units		2	2

17.3.5 Gold Room

The gold room is a separate and secure area adjoining the main process plant building. The concentrate produced by the pair of SB1350 gravity concentrators is sent by gravity to the gold room's concentrate storage tank, above the shaking table.

Several upgrades have been included in the plan for the gold room, including doubling the floor area and improving security-related items. A double-lock entrance area with security glass for video surveillance coverage is envisaged. A separate lunch room and toilet/shower room for the gold room operator is also included.

A new, enlarged shaking table package will replace the existing shaking table and associated equipment, with feed bin, overhead table magnet, peristaltic concentrate transfer pump, table tailings transfer pump, and table middlings pump. Also included are: a small Sepro SB400 batch-type gravity concentrator; a new drying oven with access lock; a new larger furnace; and a new, double-access lock safe. The highest-cost equipment items included in the new gold room package are shown in Table 17-4.

Table 17-4: Gold Room Major Equipment for 800 t/d and 1,200 t/d

Equipment		Units	800 t/d	1,200 t/d
Shaking Table	Model	1	Holman 2000	Holman 8000
Furnace	Model	1	Vcella TL80	Vcella TL225

17.3.6 Flotation

Currently, the overflow from the grinding area cyclones is first directed to a flotation area conditioning tank for addition of flotation reagents, then sent to six 10 m^3 TC10 Outotec flotation cells. With 10% air hold-up allowance, the flotation cells provide roughly 40 min of retention time at 800 t/d.

The primary flotation circuit currently comprises two rougher cells only. Rougher concentrate reports directly to the existing 4 m concentrate thickener. Primary rougher tailings report to a circuit of four rougher scavenger cells. Rougher scavenger concentrate is sent back to the conditioning tank for reprocessing.

Operationally, it is recommended to keep the flotation circuit together in one area, use the existing flotation conditioning tank that provides 4 min of retention time, and add two new 20 m^3 cells. Two new flotation cells would provide the rougher flotation capacity, while the original six 10 m^3 cells would all act as scavenger flotation cells. In addition, it is recommended to add a single cleaner-column cell. This is expected to increase the final concentrate grade from 100 g/t Au to 130 g/t Au, maintaining a similar overall flotation mass pull between 1.5% and 3%. This improved concentrate grade has positive economic benefits in reduced transport and treatment costs.

From a process design flow, the first rougher cell concentrate will combine with the cleaner concentrate and be sent to the concentrate thickener for dewatering. The second rougher concentrate will feed the cleaner-column cell and the rougher tailings will be fed to the scavenger flotation cells. The scavenger concentrate, along with the cleaner tailings, will be sent back to the flotation conditioning tank. The scavenger tailings will be sent to the tailings-thickener feed box for dewatering.

Space for the new flotation equipment would be created in the existing building by removing the existing control room to the area between the flotation and ETP areas. Handling the additional flow would require a new rougher feed pump, a tailings pump/pump box, a flotation tailings pump/pump box, and a scavenger concentrate pump; modifications to the dart valves may also be required.

An additional blower will be required for the two new 20 m^3 rougher flotation cells.

A summary of the main flotation equipment required for the flotation circuit expansion from 800 t/d to 1,200 t/d is shown in Table 17-5.

Table 17-5: Flotation Major Equipment for 800 t/d and 1,200 t/d

Equipment		Unit	800 t/d	1,200 t/d
Flotation Conditioning Tank	Conditioning Tank Working Volume	m^3	10.6	10.6
Rougher Flotation	Selected Cells		Outotec Tank Cell e10	Outotec Tank Cell e20
	No of Cells Total	#	2	2
Scavenger Flotation	Selected Cells		Outotec Tank Cell e10	Outotec Tank Cell e10
	No of Cells Total	#	4	6
Cleaner Flotation	Selected Cells		NA	Metso Microcel 0.6 x 8
	No of Cells Total	#	NA	1

17.3.7 Concentrate Thickening and Filtration

Currently, final flotation concentrate is pumped to the 4 m diameter high-rate concentrate thickener, along with filtrate return from the filtration area spillage pump. Flocculant stock solution is added to the concentrate thickener. Thickener overflow gravitates to the adjacent water tank, then is sent back to the process-water tank.

The concentrate is then thickened in the 4 m high-rate concentrate thickener to produce a 60%–65% w/w slurry underflow. The thickener underflow is then pumped to the agitated filter-feed storage tank prior to concentrate filtration. From the storage tank, slurry is pumped to a single Andritz pressure filter, with the capacity for additional plates for future expansion if required. The filter discharges into a hopper with a screw feeder below. When the hopper is full, the operator empties the concentrate cake from the hopper into bags for export.

Recent operational modifications at the process plant being undertaken during the Feasibility Study allowed for a new conveyor at the underside of the concentrate filter hopper, to direct concentrate to a stand-alone concentrate loadout structure attached to the side of the process building. In the future, loaders will load out delivery trucks.

No changes are required to the concentrate thickener and load-out facility. The existing concentrate thickener underflow pumps, filter feed tank, and concentrate filter press will be sufficient for the 1,200 t/d expansion. Using the same equipment for concentrate dewatering is possible due to a significant reduction in the flotation mass pull design criteria. The design rougher mass pull was reduced from 4.5% for the 800 t/d scenario to 2.7%, based on current operational data provided by the client. In addition, a cleaner design mass pull of 50% was considered for design.

Confirmation of the main equipment remaining for the concentrate dewatering circuit expansion from 800 t/d to 1,200 t/d is shown in Table 17-6.

Table 17-6: Concentrate Dewatering Major Equipment for 800 t/d and 1,200 t/d

Equipment		Unit	800 t/d	1,200 t/d
Concentrate Thickener	Type		High Rate	High Rate
	Model		4M Eimco Thickener	4M Eimco Thickener
	Diameter	m	4	4
Concentrate Filter Press	Type		Chamber Filter Press	Chamber Filter Press
	Model		800/50	800/50
Filter Feed Tank	Working Volume	m^3	13.3	13.3

The concentrate system is expected to allow the operation to run the filter press at the 10 to 12 cycles per shift required for the 1,200 t/d expansion. In case this scenario is not operationally feasible, Halyard recommends adding two extra plates to the existing filter press.

17.3.8 Reagents

The existing reagents area is adjacent to the process plant building and is used to make up MIBC (frother), xanthate (collector), A407 (promoter), Solenis Zetag 4105 (flocculant), and Solenis Amerfloc 485 (coagulant) flotation and thickening reagents for use in the process plant. Flocculant is mixed in a dedicated mixing/storage gravity-flow tank arrangement provided as vendor package plants. Others are dosed from drums or totes delivered ready to use. Details of the specific reagents are described next.

MIBC (frother) is delivered in drums and stored in the reagent shed until required. MIBC is dosed neat. Peristaltic-style dosing pumps deliver the reagent to the required locations within the flotation circuit. Replacement of the MIBC supply drum is carried out manually as required.

Xanthate KAX51 at 33% strength (collector) is delivered to the site in liquid form in 1,000 L totes. It is pumped neat by means of a peristaltic-style dosing pump to the flotation conditioning tank.

Cytec A407 (promoter) is delivered in drums and stored in the reagent shed until required. Cytec A407 is dosed neat. Peristaltic-style dosing pumps deliver the reagent to the required locations within the flotation circuit. (Solway recently acquired Cytec; the product will remain the same while bearing the new brand name.)

Solenis Zetag 4105 (flocculant) is delivered to site in 25 kg bags and stored in the reagent shed. A vendor-supplied mixing and dosing system was installed, including a flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powder flocculant is loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant is pneumatically transferred into the wetting head, where it is contacted with water. Flocculant solution, at 0.25% (w/w) is agitated in the flocculant mixing tank for a pre-set period, after which the flocculant is transferred to the flocculant storage tank using the flocculant transfer pump. Flocculant will be dosed to both tailings thickeners, and to the concentrate thickener, using variable-

speed progressive cavity hose-style pumps. Flocculant will be further diluted to approximately 0.025% w/w just prior to the addition point.

Solenis Amerfloc 485 (coagulant) is delivered in 1000 L totes and stored in the reagent shed until required. Coagulant is dosed neat. Peristaltic-style dosing pumps deliver the reagent to the tailings thickener. Replacement of the coagulant supply drum will be carried out manually as required.

Reagent manufacturers and consumption rates were updated and escalated based on current operational data. The use of flocculant and coagulant for tailings dewatering was considered for the 1,200 t/d expansion based on recent testwork results, which has confirmed that the tailings sample can achieve 72.6% solids w/w at the tailings thickener underflow, while using 37 g/t AMERFLOC 487 coagulant and 10 g/t ZETAG 4105 flocculant at a solids loading rate of 0.45 t/(m^2•h). In addition, testwork done for the rougher concentrate sample has shown that the material can achieve 80.1% solids w/w while using 20 g/t ZETAG 4105 flocculant at a solids loading rate of 0.08 t/(m^2•h).

The existing xanthate preparation plant used for the 800 t/d design will not be used going forward, as the reagent is now bought in liquid form. During the realization of this study, new storage tanks were being sized by the client's operations team for xanthate, coagulant, frother, and promoter. The 1,200 t/d expansion Project will make use of these new tanks, which will be sufficient for the additional flow of reagents required for the expansion. The existing flocculant preparation plant will be used for the 1,200 t/d expansion. One additional dosing pump for xanthate, frother, promoter, and flocculant will be required.

The existing dedicated sump pump located in this area will be sufficient for clean-up purposes.

A summary of the main reagents types and equipment required for 800 t/d and 1,200 t/d are shown in Table 17-7.

Table 17-7: Reagent Type and Major Equipment for 800 t/d and 1,200 t/d

Equipment		Unit	800 t/d	1,200 t/d
Frother	Name		MIBC	MIBC
	Transport Packaging		Drums 161 kg	Drums 161 kg
	Storage Tank Volume	m^3	0.2	1.2
Collector	Name		KAX51-33% Strength	KAX51-33% Strength
	Transport Packaging		1,000 kg totes	1,000 kg totes
	Storage Tank Volume	m^3	0.8	1.6
Promoter	Name		Solvay A407	Solvay A407
	Transport Packaging		1,075 kg totes	1,075 kg totes
	Storage Tank Volume	m^3	0.4	1.2
Flocculant	Name		Solenis Zetag 4105	Solenis Zetag 4105
	Transport Packaging		25 kg bags	25 kg bags
	Mixing Tank Volume	m^3	0.5	0.5
	Storage Tank Volume	m^3	0.5	0.5
Coagulant	Name		Solenis Amerfloc 485	Solenis Amerfloc 485
	Transport Form		1,000 L totes	1,000 L totes
	Storage Tank Volume	m^3	NA	1.2

17.3.9 Tailings

At the 800 t/d throughput, the tailings from the flotation circuit are designed to report to a 12 m high compression thickener to produce a 65%–70% w/w slurry before being pumped to the paste plant or the tailings storage facility as required.

The existing tailings thickener overflow currently gravitates to the adjacent process-water tank. This along with the pumped concentrate thickener overflow, any excess bulk circuit water, and make up tailings decant water make up the process water. This process water is distributed by means of the process-water pump to the grinding circuit, flotation roughers, scavengers, concentrate thickener, paste plant, and plant service points.

A vertical-spindle sump pump located in the thickener sump returns spillage to the tailings thickener feed box.

At the 1,200 t/d throughput, additional flotation tailings thickening capacity will be required. Using similar solids loading assumptions for the additional 400 t/d, a new 9 m diameter high-compression type thickener will be added to the process, located adjacent to the existing thickener. Tailings coming from flotation will be fed into a new tailings thickener feed box, which will distribute the flow to both tailings thickeners by gravity.

The existing tailings thickener will be fed at a rate of 34.2 t/h. The underflow will be sent typically to the paste plant's feed tank as per existing arrangement. When the paste plant is not operating, the filter feed tank, when full, will overflow back to the existing tailings pump box and be sent to the North TMF.

The new tailings thickener will be fed at a rate of 18.4 t/h. New pumps will be required to convey the new tailings thickener underflow to a new pump box, then to a new tailings pump, then a new insulated and heat-traced high-density polyethylene (HDPE) pipeline will convey the tailings to the North TMF.

A summary of the main equipment required for tailings dewatering for 800 t/d and 1,200 t/d is shown in Table 17-8.

Table 17-8: Tailings Dewatering Major Equipment for 800 t/d and 1,200 t/d

Equipment		Unit	800 t/d	1,200 t/d
Tailings Thickener #1	Description		12 M Eimco Thickener	12 M Eimco Thickener
	Duty		High Compression	High Compression
	Diameter	m	12	12
Tailings Thickener #2	Description		NA	9 M Eimco Thickener
	Duty		NA	High Compression
	Diameter	m	NA	9

17.3.10 Services

A second process-water tank will be required to handle increased loads. The overflow from the existing process-water tank would feed the new process-water tank as well as the overflow from the new tailings thickener. A new process-water pump would also be required to allow supply to the new milling circuit.

A new gland water pump will be required for the 1,200 t/d expansion.

The existing clean-water tank, clean-water pump and fire water pumps will be used for the 1,200 t/d expansion. Fresh water will continue to be supplied from the well pump, which will deliver the fresh water to the clean/fire-water tank.

Fresh water will be used for the following duties:

- Low-pressure clean water for in-plant use
- Reagent make-up via the raw-water pump
- Supply to the gold room for gold tabling
- Supply to the paste plant filtrate feed tank.

In addition, one new air compressor, dryer, and receiver will be required for the new cleaner flotation column cell and for plant instrumentation air.

A summary of the main equipment required for process services for 800 t/d and 1,200 t/d are shown in Table 17-9.

Table 17-9: Services Tankage Sizes for 800 t/d and 1,200 t/d

Equipment		Unit	800 t/d	1,200 t/d
Services	Process Water Tank Working Volume (Existing)	m^3	169	169
	Process Water Tank Working Volume (New)	m^3	NA	100
	Clean/Fire water tank working volume	m^3	260	260

17.3.11 Energy Requirements

The anticipated additional loads associated with process plant and the paste plant changes means that the energy requirement is expected to increase by approximately 2,150 kW of installed power.

The incoming 25 kV power line from Hydro One designed for 6.8 MW has sufficient available spare capacity to power the process and paste plant expansions. A 5/7.5 MVA, 25–4.16 kV transformer supplies the process plant building and other minor auxiliary surface loads.

The process plant building is currently supplied from a 4.16 kV single overhead line circuit. The existing 4.16 kV overhead distribution line conductors currently supplying the process plant building are undersized to power the proposed mill expansion. Power required for the operation of the mill and paste plant expansion will be provided from a spare circuit breaker located in the main substation 4.16 kV switchgear.

The 4.16 kV single-circuit overhead distribution line supplying the mill building will be modified to a double-circuit line complete with new riser, dead-end structure, and pole-mounted fused disconnect switch. If the existing overhead distribution line cannot be modified, a new overhead line will have to be constructed.

Underground cables from the new 4.16 kV overhead line dead-end structure will be installed in a trench and terminated in a new modular switchgear building. The new switchgear building will contain the 4.16 kV distribution switchgear, 600 V motor control centres, ball mill variable-frequency drive (VFD), auto-transfer switch, and switchgear-building auxiliary services.

The 4.16 kV motor control centre will feed the 4.16 kV loads and a 4.16–0.6 kV step-down transformer. A 1.5 MVA, 4.16–0.6 kV outdoor liquid-filled transformer will step down the distribution voltage from 4.16 kV–600 V. The 4.16 kV and 600 V distribution equipment will have metering capacity for power reading and power management.

A self-contained diesel generator, skid-mounted on gravel, will be provided to supply power to the emergency and critical power loads. The generator will be provided with a 12 h sub-base fuel tank. In the event of a utility power outage, the generator will provide power to critical equipment, such as life and safety systems, communications equipment, security equipment, and control systems. The generator loads and size will be finalized during the detailed design.

Process-critical network infrastructure will have redundant communications paths and be powered by uninterruptible power supplies (UPS).

The UPS will provide backup power to critical control systems including process control. The UPS will be sized to permit operations to shut down, and back up the computer and control systems, to facilitate start-up on restoration of normal (utility) power.

18 PROJECT INFRASTRUCTURE

18.1 Site Description

The general site layout plan with the existing processing plant, administrative buildings, and water management ponds are shown in Figure 18-1. Figure 18-2 shows the process plant complex along with the proposed expansion. The general site location with existing North TMF and proposed South TMF is also shown in Figure 18-3.

The North TMF is located within a valley immediately north of the processing plant and administrative buildings. The North TMF is confined to the north and south by bedrock valley walls. The remaining perimeter is confined by embankment dams, namely the northwest and southwest saddle dams at the west side and the east dam at the east side. the existing site water management ponds (east containment pond and expansion pond) are located east of the processing plant and administrative buildings. mine waste rock is currently stockpiled between the processing plant and the North TMF.

The proposed South TMF is positioned in a low-lying basin approximately 2 km south of the mine site. The proposed South TMF is contained by ridges and bedrock features on the east, south and west sides and by the proposed north dam on the north side.

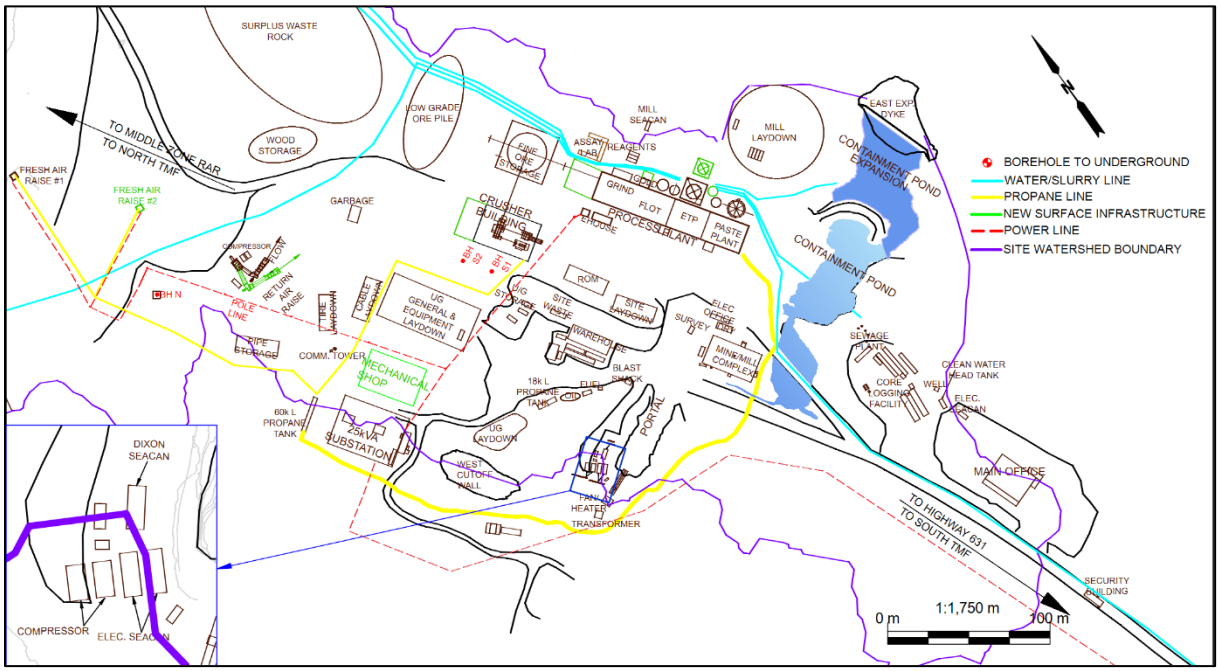


Figure 18-1: Sugar Zone Mine Site with Expected Future Infrastructure

18.2 Site Infrastructure

18.2.1 Site Access Roads

The site is accessible via the 20 km all-weather road network of the White River Forest Sustainable Forest License (SFL) area that connects to Highway 631 and from there to Ontario Highway 17. The site access road was upgraded in 2018 with a Granular A base to support heavy truck traffic to the site. Harte Gold collaborates with the holder of the SFL regarding road maintenance. In 2020, the site access road was upgraded with guardrails in narrow or elevated sections to provide an additional level of protection for road users.

18.2.2 Power Supply

Electric power is provided by a 45 km, 25 kV wooden pole power line that parallels the road corridors from the mine site to the White River water treatment plant on Highway 631 north of the town of White River.

This 25 kV power line is connected to the site substation and is transformed to 13.8 kV and 4.16 kV by two transformers with 5 MVA capacity. Each transformer provides power to two separate switchgear lineups.

The 13.8 kV switchgear lineup feeds:

- Portal infrastructure:
  - Air Compressors
  - Ventilation fans (upper ramp)
  - Underground infrastructure
  - Sugar Zone—South, North, and Middle Zones.
- Surface infrastructure:
  - Admin/office facility, dry facilities, warehouse, and coreshack
  - Maintenance facility
  - Security/guardhouse
  - First-aid facility
  - Roadway lighting.

The 4.16 kV switchgear lineup feeds:

- Mill infrastructure:
  - Mill facility
  - Ore crushing
  - Concentrator
  - Effluent treatment plant

  - Paste plant
  - Sewage treatment plant.
- Surface ventilation:
  - Main fresh air fans
  - Main return air fans.

A CAT back-up generator is capable of supplying 230 kW emergency electricity to the process plant.

18.2.3 Processing Plant Buildings and Facilities

The current process plant consists of several structures including:

- Crusher building
- Fine ore stockpile building
- The main mill building houses:
  - Grinding circuit
  - Flotation circuit
  - ETP—See Section 18.4-Water Infrastructure
  - Paste backfill plant.
- Electrical room
- Gold room
- Water and material storage:
  - Firewater tank
  - Process water tank
  - 12 m diameter high compression tailings thickener
  - Cement silo
  - Filter feed tank.
- Material movement conveyor belts and pipelines.

Expansion will require the following changes to the process plant footprint to accommodate the modifications described in Section 17:

- Crusher building expansion of 338 m^2
- Mill building expansion of 224 m^2
- Electrical room expansion of 32 m^2
- Gold room expansion of 60 m^2
- Additional 9 m diameter tailings thickener
- Additional process water tank.

All recommended modifications to recovery circuits, paste backfill plant, and ETP will fit within these expanded footprints. The current and expanded footprints are shown in Figure 18-1.

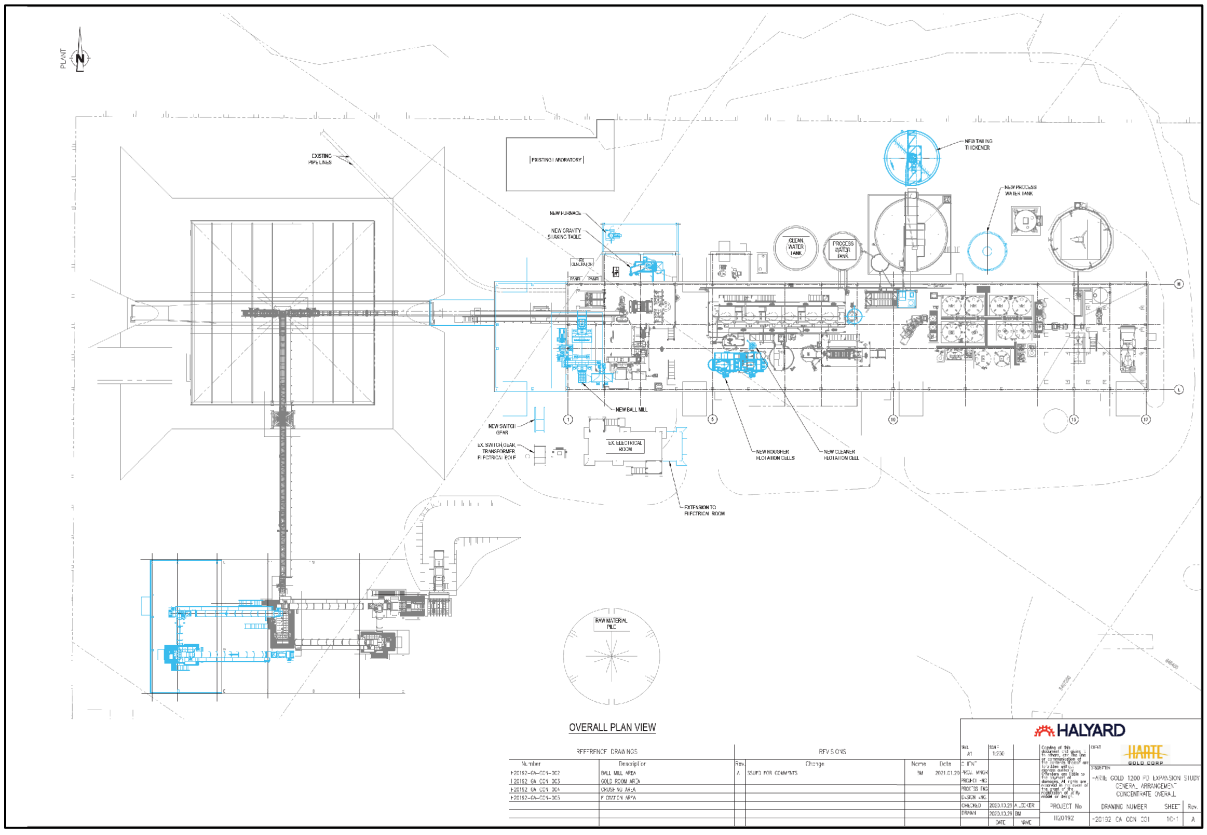


Figure 18-2: Process Plant Complex Showing Additions in Blue

18.2.4 North Tailings Management Facility

Initial development at the North TMF consisted of the Stage 1 east dam and northwest and southwest saddle dams, which were constructed to El. 405 m in 2019. An interim 1 m raise of east dam and West Saddle Dams was constructed in 2020. The interim raise consisted of fill material placement over the crest at El. 405, with reduction of the crest width from approximately 6.5 to 3.0 m. The Emergency Spillway was also raised as part of the interim east dam raise.

Harte implemented a change to the tailings processing operation in early 2020, which involved producing filtered tailings using the existing paste backfill plant with added filters. The filtered tailings have been stacked at the south hillside of the North TMF since early 2020. The future modifications to this TMF are described in detail in Section 18.3.

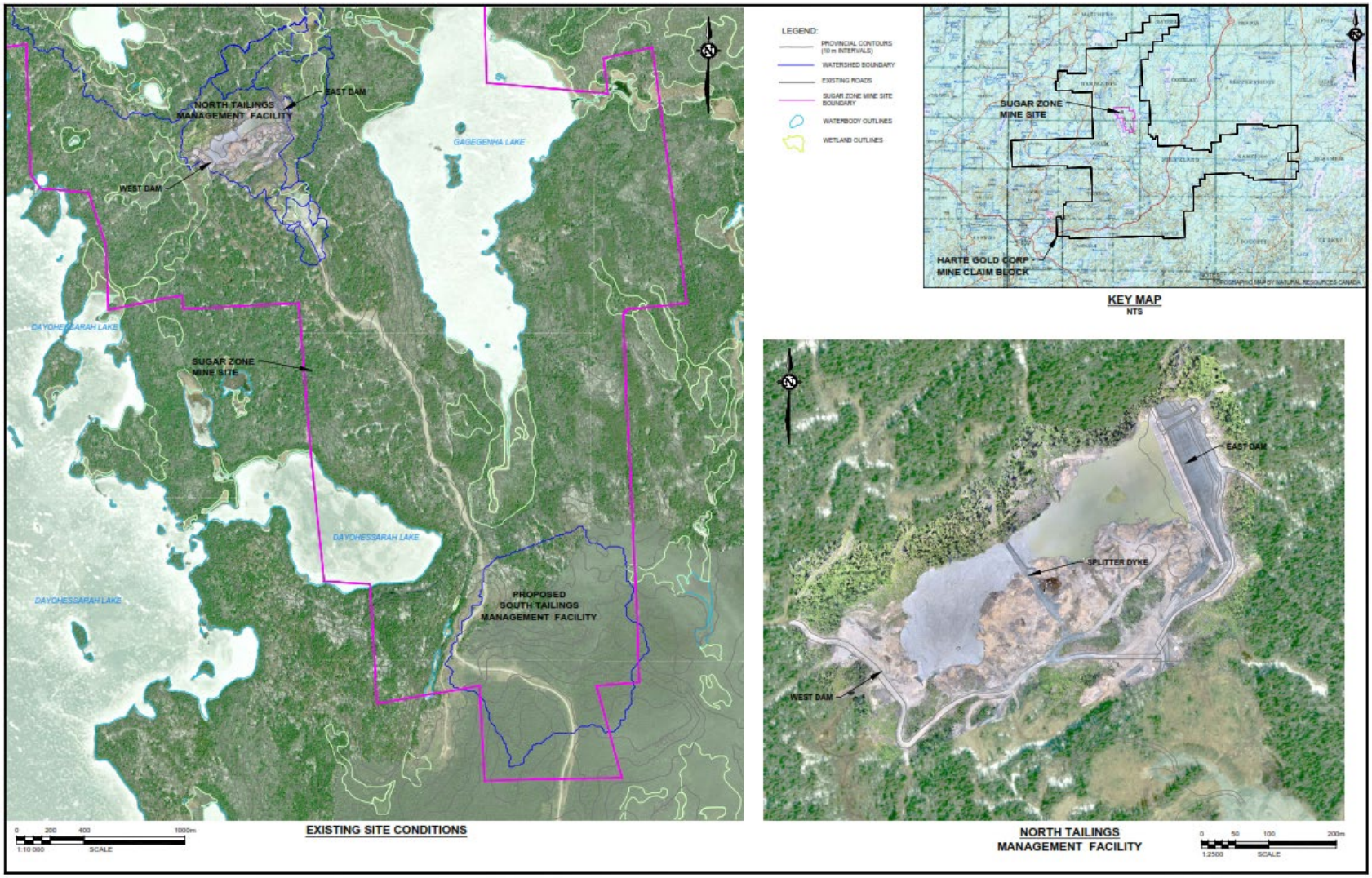


Figure 18-3: Existing Site Plan Showing South TMF Location (Wood, 2021b)

18.2.5 Ancillary Buildings

Ancillary buildings are generally portable and/or modular structures.

18.2.5.1 Gatehouse

The gatehouse is a prefabricated, portable trailer. Base skirting is plywood and/or metal, with metal roofing. Water pipe(s) are copper or plastic. Foundations are wood timbers and/or modest concrete footings over top of a competent aggregate pad.

18.2.5.2 Core Shack and Core Storage

The coreshacks consist of two prefabricated portable trailers.

18.2.5.3 Assay Lab

The assay lab is a pre-fabricated composite facility housed in portable, connected trailers. One trailer is dedicated to sample preparation, a second is dedicated to fire assaying gold and silver; while the third trailer is dedicated to wet chemistry, gravimetric finish equipment, instrumentation, and an office. Electricity, propane, and water distribution, dust collection, fume scrubbing, and air conditioning and filtration are important attributes of the assay lab.

18.2.5.4 Mine Surface Facilities

Refer to Section 16.19 for detail around site infrastructure that serves the underground mine.

18.2.5.5 Modular Office and Dry Trailers

Containers and trailers have been installed on competent aggregate pads. Base skirting is plywood and/or metal, with metal roofing. Water and domestic sewage pipes are heat-traced copper or plastic. Foundations are wood timbers and/or modest concrete footings over top of a competent aggregate pad.

18.2.6 Storage Tanks

Storage tanks are listed below.

- Storage tanks for firewater (<150 m^3). As per insurance underwriter requirements, containment pond water is an alternate firewater source, eliminating the freezing challenges of surface tanks.
- Fuel (propane, diesel) storage containers (<50 m^3 each for diesel, <100 m^3 each for propane) are strategically located to supply fuel to the process plant back-up generator and heating units at the mine site.
- Potable water (<25 m^3) is stored in tanks that are inside the containerized potable water treatment plant.

- The mine site diesel requirements are stored in one 25,000-L fuel tank as described in Section 16.19.

18.2.7 Pipelines

The significant site pipelines are listed below (internal process plant piping is excluded).

- Steel pipeline with Victaulic fittings to transmit mine water to the Containment Pond.
- HDPE pipeline (10 cm diameter) to transmit water from the Containment Pond to the TMF.
- HDPE pipeline (10 cm diameter) to transmit fresh water from Gagegenha Lake to the process plant. This line is bi-directional and also used as an effluent discharge line.
- Heat-traced pipelines (5 cm diameter) to transmit service water throughout the site.
- Polyethylene pipeline (10 cm diameter) to transmit domestic sewage to the sewage treatment system.
- Two HDPE pipelines (10 cm diameter) to transmit water from the TMF to the process plant.
- HDPE pipeline (10 cm diameter) to transmit tailings from the process plant to the TMF.
- Steel compressed air lines, with Victaulic fittings to distribute compressed air from the main compressor to the mine and to the process plant.
- Copper and steel lines to deliver propane to building and mine air heaters.
- Steel or high strength polymer lines to distribute firewater to firehose cabinets.
- Upon commissioning of the South TMF two heat-traced HDPE pipelines (10 cm diameter) will be extended from the process plant complex to the South TMF location (~2 km) to transport slurry tailings to the South TMF and return reclaim water to the effluent treatment plant.

18.2.8 Communications

A microwave tower provides a communication link to White River. The operation has an internet connection with speeds of up to 40 Mbps, upload and download.

18.3 Life-of-Mine Tailings Management Facility Design

18.3.1 Site Conditions and Background Information

18.3.1.1 Topography and Watershed

The site is located between Gagegenha Lake and Dayohessarah Lake, which are northeast and southwest of the existing North TMF valley. The site is located within Watershed 2BC-11, within the Lake Superior watershed, approximately 10 km from the watershed divide between the Great Lakes (Atlantic) and Arctic watersheds. Both the North TMF and proposed South TMF watersheds drain into the Gagegenha Lake watershed.

The dominant landform at the site is shallow bedrock outcrop overlain with subdominant till ground moraine and/or organic deposits. The topography is moderate to rugged and well drained, with elevations ranging from 390 masl to 480 masl at the North TMF and 380 masl to 410 masl at the proposed South TMF. Vegetation is boreal with jack pine, fir, poplar, and birch occupying the uplands, and cedar, tamarack, and spruce in the poorly draining lowlands (Buck, Hayden, & LeBlanc, 2012).

18.3.1.2 Regional Hydrology

The climate at the site is boreal, with hot summers and cold, snowy winters. The daily average temperatures vary from +15.3°C in the summer to -14.0°C in the winter. The ground is usually snow covered from December through to the end of April. Annual average precipitation for the site was determined to be 969.6 mm. This information is based on data obtained from Environment Canada Climate Normals for Wawa, Ontario, from 1981 to 2010. Hydrometeorology data from nearby meteorological stations applicable to the site are summarized below:

- Average annual precipitation—969.6 mm (Wawa, Environment Canada)
- 100-year 30 d rain + snowmelt—432.6 mm (White River, Hydrometeorology Division, Canadian Climate Centre);
- 100-year 24 h rainfall—115.2 mm (Wawa IDF, Environment Canada)
- 1,000-year 24 h rainfall—154.6 mm (Extrapolated according to Gumbel Extreme Value distribution)
- Regional Storm—193 mm in 12 h (Timmins storm, Ontario Ministry of Natural Resources 2002)
- PMF—379 mm in 24 h (based on Hershfield Method from Pukaskwa and Wawa rainfalls, Environment Canada)
- Mean annual lake evaporation—515 mm/a (Environment Canada data for Rawson Lake, Ontario (1969 to 1999), Kemptville, Ontario (1968 to 1995) and Ottawa CDA, Ontario (1962 to 1998).

18.3.1.3 Site Seismicity

The site is situated at approximately 48.80° N latitude, 85.01° W longitude within the “Stable Canada,” which is the lowest seismic zone in Canada. According to the available historical earthquakes data, twenty earthquakes with magnitude 3.0 and greater have occurred within 300 km of the site over the past century. The largest recorded historical earthquake, a magnitude 4.2 event, occurred on 7 December 2006, about 250 km from the site. The peak horizontal ground accelerations (PGA) corresponding to Site Class C, obtained from the National Building Code of Canada (NBCC, 2015), for the site are summarized Table 18-1.

Table 18-1: Summary of Peak Ground Accelerations

Probability	40% / 50 year	10% / 50 year	5% / 50 year	2% / 50 year
Return Period (years)	100	475	1,000	2,475
PGA—Firm Ground	0.003	0.011	0.019	0.036

Source: NBCC, 2015.

18.3.1.4 Subsurface Conditions

Subsurface soil conditions at the site comprise competent bedrock, which is generally overlain in some areas by a thin veneer of silt, reported to range in thickness from areas with bedrock at surface to about 5 m. Groundwater flows from areas of high hydraulic head to areas of low hydraulic head. Based on the observed groundwater surface elevations at the site, groundwater flow directions in the upper unconfined unit (i.e., overburden and upper weathered bedrock) generally reflect that of topography and surface water. Detailed descriptions regarding groundwater flow and hydrogeology of the site can be found in the reports prepared by Amec Foster Wheeler in 2016 (Amec Foster Wheeler, 2016a, 2018).

The general subsurface conditions encountered at the North TMF and the proposed South TMF are summarized in the following sections based on geotechnical investigation reports for both the North TMF (Amec Foster Wheeler, 2016a, 2016b) and South TMF (Wood, 2021a). Additional geotechnical investigations should be carried out to support and optimize detailed designs of North TMF and South TMF.

Subsurface Conditions at North Tailings Management Facility

Several geotechnical and hydrogeological site investigations were conducted between 2012 and 2016 at the North TMF site with the latest investigation carried out by Amec Foster Wheeler (2016a, 2016b). These investigation programs included test pits, boreholes and hydrogeological testing to investigate the geotechnical subsurface conditions. Based on these site investigations, the overburden at North TMF generally comprises the following successive soil strata:

- Organic topsoil
- Cobbles and boulders with some sand
- Silty sand to sandy silt.

The majority of test pits and trenches encountered a surficial layer of dark brown, dry to moist, organic topsoil. The thickness of this layer ranged between 0.03 and 1.2 m and it is expected to vary in thickness and quality across the site. The surficial organic layer was generally underlain by silty sand to sandy silt layer. At some locations, the surficial peat layer was underlain by a layer of predominantly cobbles and boulders with some sand. The cobbles and boulders layer was generally dry to wet, grey, typically paired with seepage at ground surface, and early refusal. This material was encountered to a depth ranging between 0.1 to 4.6 m below ground surface.

The silty sand to sandy silt stratum was encountered below the organic layer in most of the test pits and boreholes. This material continued up to the limits of the excavator reach at 4.6 m below grade,

or until the bedrock contact. This material was a brown or red/brown or grey material, ranging from a silty sand to sandy silt, to occasionally silt. The material typically contained some to trace gravel and trace clay, and occasionally cobbles and boulders. Soil samples were generally moist to wet. Grain size distribution analysis of select samples of this material indicated 0% to 28% gravel, 40% to 60% sand, and 25% to 42% fines. Natural moisture content was measured to range from 12 % to 44 %. Direct shear and Consolidated Undrained Triaxial tests were also conducted to assess the shear strength characteristics of the sandy silt to silty sand layer.

Seepage was encountered in most of the 2016 test pits (Amec Foster Wheeler, 2016b). Short-term water seepage level recorded in the test pits and trenches ranged between the surface and 2.4 m below existing grades, with most of the test pits encountering groundwater seepage at 0.9 m below grade.

Based on the estimated hydraulic conductivity values during the 2016 investigation, the upper weathered bedrock is hydraulically connected to the overlying overburden and organic deposits with values in the 10^{-5} to 10^{-6} m/s range. The deeper bedrock, beyond 5 m depth, indicates less permeable condition with hydraulic conductivity values ranging from 10^{-8} to 10^{-9} m/s (Amec Foster Wheeler, 2016b).

Subsurface Conditions at Proposed South TMF

A geotechnical investigation was carried out in October 2020, and included boreholes, monitoring wells, dynamic cone penetration tests (DCPT), test pits (TP), approximately 500 m of test trenching (TT), and Packer testing in boreholes. The details and results of the investigation program can be found in the geotechnical site investigation report (Wood, 2021a). Previously, DST Consulting Engineers Inc (DST) had carried out test pits at the proposed South TMF site (DST, 2019). Based on these site investigations, the following successive soil strata were encountered at the proposed South TMF location:

- Gravelly sand
- Sand with traces of organics and gravel
- Sand and silt to silty sand
- Silt with traces of sand and gravel.

The SPT N-values determined within the gravelly sand stratum indicated medium to very dense state of compactness, except in one borehole where it indicated loose to medium dense compactness. The SPT N-values within the sand and silt to silty sand stratum generally indicated medium to very dense state of compactness, except in one borehole where borehole where it indicated loose conditions. The sand and silt to silty sand and gravelly sand deposit varies in thickness from negligible (in areas with bedrock at surface) up to 20 m. Notably, thick deposit was observed at the proposed South TMF Dam's left abutment area.

A thin to moderately thick layer of silt with trace of sand and gravel overlies bedrock in some areas.

Similar to the North TMF site, groundwater flow directions in the upper unconfined unit (i.e., overburden and upper weathered bedrock) generally reflect that of topography and surface water. Groundwater table within the proposed South TMF dam footprint was encountered at less than 1 m above the bedrock or not encountered, except in one borehole located at valley bottom where the groundwater table was encountered at 4.7 m above bedrock surface. Results of Packer testing carried out during the 2020 investigation (Wood, 2021a) found bedrock hydraulic conductivity to be of the order of 10^{-6} m/s at shallow depths and decreasing to 10^{-7} m/s generally at about 5 to 6 m depth.

18.3.1.5 Tailings Characteristics

Tailings is generated from conventional and flotation recovery processes. The current mill throughput is approximately 800 t/d, and will increase to 1,200 t/d by 2023. The tailings solids have a specific gravity of 2.84.

The grain-size distribution analysis tests completed on the tailings samples indicated 25% sand, 69% silt, and 6% clay size composition. Based on tailings settling tests, settled dry density of the slurry tailings is assumed to be 1.35 t/m^3. Standard Proctor compaction tests carried out on filter tailings indicated standard Proctor maximum dry density (SPMDD) of 1.72 kg/m^3.

The tailings have benign geochemistry, with low sulphur, low sulphide, and is non-potentially acid generating (non-PAG) with no metal leachate based on the geochemical characterization report by Amec FFW (2016c).

18.3.2 Tailings Management Facilities—Classification and Design Criteria

18.3.2.1 Dam Classification

The consequence classification of the North TMF and South TMF dams was carried out based on the standards-based approach outlined by the Canadian Dam Association (CDA, 2013 and 2019) *Dam Safety Guidelines* as well as the *Global Industry Standard on Tailings Management* (GISTM, 2020). The *Lakes and Rivers Improvement Act* (LRIA) guidelines (MNR, 2011) also provide criteria to determine the Hazard Potential Classification (HPC) for dams.

The HPC for the North TMF and South TMF dams is classified as “High” in accordance with the LRIA considering potential environmental losses in the event of dam breach. The corresponding classifications under CDA (2013 and 2019) and GISTM (2020) are also “High” for the North TMF and South TMF dams based on environmental consequences. Design criteria for the North TMF and South TMF were developed based on the classification of High (LRIA, 2011).

Detailed discussion of the dam classification, selection of design events and criteria can be found in the Feasibility Study Report (Wood, 2021b).

18.3.2.2 Environmental Design Flood

The environmental design flood (EDF) is defined as the design flood event for the site water-management system to prevent discharge of untreated water to the environment. The 1-in-100-year flood event was selected as the EDF, which is consistent with standard practice in Ontario.

18.3.2.3 Inflow Design Flood

The Inflow Design Flood (IDF) is defined as the largest runoff event that a facility is designed to discharge safely (e.g., through spillways) and prevent overtopping of the water or tailings containment structures and dams.

For the High classification, the LRIA guidelines (MNR, 2011) recommend that the IDF is to be taken as the 1,000-year flood or Regulatory Flood, whichever is greater, to 1/3 between the 1,000-year flood and Probable Maximum Flood (PMF). The IDF for the project dams was adopted as 1/3 between the 1,000-year flood and the PMF. This is equivalent to the CDA (2019) requirement during the operations phase and satisfies GISTM (2020) design flood requirement for operational phase. GISTM (2020) design requirement for post-closure condition may be considered at the time of site closure.

18.3.2.4 Design Earthquake

Design earthquake criteria based on dam consequence classification are provided by CDA (2013, 2019), MNR (2011), and GISTM (2020). For High-consequence dams in the operations phase, CDA (2013, 2019), MNR (2011), and GISTM (2020) requirements are generally the same, with consideration of the 1-in-2,475-year earthquake. GISTM (2020) requires all tailings dam to have closure designs that can withstand the 1:10,000-year design earthquake. Therefore, the 1-in-10,000-year design earthquake was adopted for both North TMF and South TMF dams to minimize potential future upgrade requirements.

The PGA value for the 1:10,000-year design earthquake event was obtained by extrapolation method as described by NBCC (2015). The extrapolated design PGA values corresponding to 10,000-year seismic event is 0.1 g. It is noted that NBCC (2015) suggests that the extrapolated values be used as preliminary and recommends site specific seismic hazard study to evaluate 10,000-year seismic hazard.

18.3.2.5 Summary of Design Criteria and Considerations

Table 18-2 presents a summary of the design criteria and considerations for the design of North TMF and South TMF. Tailings storage requirement/schedule are summarized in Section 18.6.

Table 18-2: Summary of Design Criteria and Considerations

Item	Design Criteria
Project Location	35 km NE of White River, Ontario; UTM Zone 16N; NAD 83
Mine Production	Total Expected Ore Milled = 3.5 Mt
	Mill Throughput = 800 t/d, increasing to 1,200 t/d by 2023

Item	Design Criteria
	Mine Life = 9 years (through 2029)
Site Constraints	Gagegenha Lake Fish Habitat downstream of the North TMF and South TMF to be protected, minimum 60 m setback from shoreline
	Development cannot impact the Dayohessarah Lake Watershed
	Release of treated water to the environment (Gagegenha Lake) to be maximum of 4,100 m^3/d to manage the EDF
	Period of discharge to the environment = May 15 to November 30
	Geometry of the North Valley limits the crest elevation for the North TMF east dam
Construction Requirement	Minimum dam crest width = 6 m
	LOM tailings production = 3.402 Mt
Tailings Properties and Tonnages	Tailings used as paste backfill over mine life = 0.865 Mt
	Tailings used for selective flotation mass pull over mine life = 0.52 Mt
	Total tailings tonnage requiring storage = 2.537 Mt
	Conventional Thickened Slurry Tailings parameters: Medium term average in-situ tailings density = 1.35 t/m^3 Void ratio of deposited tailings = 1.10 Discharge tailings slurry solids content = 60%
	Filtered Tailings parameters: Average compacted dry density = 1.7 t/m^3 Void ratio of deposited tailings = 0.65
	Specific gravity of solids = 2.84
	Tailings Geochemistry = Low sulphur, low sulphide, non-acid generating (Non-PAG) and non-metal leaching
Operational Criteria	Maximum Discharge of Process Water: as per water balance
	Supernatant water from the TMF recycled for re-use in mill processing
	Water to be treated before being released to the environment (Gagegenha Creek)
Dam Classification (HPC)	North TMF Dams: High
	South TMF Dam: High
	Emergency Spillway Required = Yes
	Emergency Spillway to Discharge towards Gagegenha Lake and Creek
Hydrology Criteria	EDF: 1-in-100-year critical hydrologic event (30-d Rain on Snow or 24-hour Rainfall)
	Inflow Design Flood (IDF): 1/3 between the 1,000-year flood and the Probable Maximum Flood (PMF) = 229.4 mm
Freeboard	Sufficient for full containment of the IDF Peak Flow + Design Wave Run-Up
	Design wave run-up = no overtopping by 95% of the waves caused by the most critical wind with a frequency of 1:1,000-year when reservoir is at its maximum normal elevation.
Design Earthquake	1 in 10,000-year return period design earthquake, PGA = 0.1 g
Slope Stability Criteria	Operation and Closure – Static Loading: - End of Construction (Perimeter Dams): Factor of Safety ≥1.5 - End of Construction (Internal Structures): Factor of Safety ≥1.3 - Steady State: Factor of Safety ≥1.5 - IDF Loading: Factor of Safety ≥1.3
	Operation and Closure – Earthquake Loading - Pseudo-static: Factor of Safety >1.0, or acceptable deformation - Post-earthquake: Factor of Safety ≥1.2
Seepage Criteria	The perimeter dams including their foundations are to be designed as low permeability structures.
	Seepage collection ditches and pond will be required if seepage is encountered at the downstream toe of the dams. The seepage collection facilities to fully contain the flow without overflowing.
	Collected seepage to be pumped back to North TMF and South TMF.

Item	Design Criteria
Native Overburden	Consists of sand and gravel, silty sand, and silt strata. Occasional boulders and cobbles also present (Amec Foster Wheeler, 2016b)
Closure Criteria	The North TMF and South TMF will be dewatered by treating the water and discharging it to the environment and also pumping water to underground workings to expedite flooding.
	North TMF and South TMF surfaces to be covered with locally available overburden as necessary and re-vegetated.

Item	Design Criteria
TMF Dam Characteristics	Embankment dam with low permeability element and treated foundation to minimize seepage to Gagegenha Lake
	Impervious upstream bituminous geomembrane (BGM) liner of the embankment dam founded on treated bedrock; remaining embankment to be founded on competent overburden or bedrock.
	Location of North TMF Emergency Spillway = east dam
	Location of South TMF Emergency Spillway = north dam

18.3.3 Tailings Deposition Plan and Construction Staging

A trade-off study was performed to establish North TMF and South TMF tailings storage requirements and dam raise staging for the current LOM tailings tonnage and production schedule (Wood, 2020a, 2020b). The tailings storage requirements at the North TMF and South TMF based on the current LOM are summarized in Table 18-3. Detailed tailings deposition plans for North TMF and South TMF can be found in the Feasibility Study Report (Wood, 2021b).

Table 18-3: Tailings Storage Requirements at North TMF and South TMF

Tailings Storage Requirements	Tonnage[1]	Volume (m^3)[2]
North TMF—Filtered Tailings	1,112,000	646,000
North TMF—Slurry Tailings	204,000	151,000
Total North TMF	1,316,000	797,000
South TMF—Slurry Tailings	1,222,000	905,000
Total South TMF	1,222,000	905,000
Total North TMF and South TMF	2,537,000	1,702,000

Notes [1] Some totals do not add due to number rounding.
[2] Volumes based on LOM and do not include water storage volumes

The existing North TMF dams are to be raised in stages to allow operation at North TMF until April 2025. The tailings deposition strategy envisioned for the North TMF is summarized as follows:

- A new splitter berm is to be constructed east of the existing splitter berm. Initial slurry deposition is to take place from the existing splitter berm with the objective to push the reclaim pond eastward. The splitter berm is to be raised concurrently with slurry deposition elevation.
- Initial filter tailings placement to be started from the West Dam and sloped from west to east.
- Filter tailings placement is to extend (in stages) onto the slurry beach to extend the footprint of the filter tailings eastward.
- Slurry spigots will be progressively moved east along with the filter tailings to allow the filter tailings footprint to expand to accommodate the greater filtered tailings volume.
- Subsequent slurry deposition will be from the eastern slope of the filter tailings.
- Ultimate filter tailings will be placed to El. 414.6 m against the West Dam at a 0.5% slope.
- Ultimate slurry tailings deposition will be from the filter tailings at El. 411.4 m at 1% slope.

South TMF will be developed in 2024 with tailings deposition transitioning from North TMF to South TMF in 2025. Tailings deposition at South TMF will be by spigotting of slurry tailings from the South TMF north dam.

Table 18-4 provides a summary of the proposed North TMF and South TMF dam raise stages for the current LOM. Each raise was targeted to provide capacity for at least two years of tailings production. Intermediate dam stages may be revised at detailed design to suit any change to the LOM or other operational considerations.

Table 18-4: Annual Tailings Deposition and Dam Construction Schedule

Year	North TMF					South TMF	
	Incremental Filtered Tailings Volume Placed (m^3)	Incremental Slurry Tailings Volume (m^3)	West Dam Raise Crest El. (m)	East Dam Raise Crest El. (m)	Splitter Berm Raise El. (m)	Incremental Slurry Tailings Volume (m^3)	North Dam Raise Crest El. (m)
2021	151,793	47,333	411.6	410.0	403.4	-	-
2022	135,390	40,556	-	-	405.5	-	-
2023	143,864	27,037	415.0	412.6	408.0	-	-
2024	164,156	27,111	-	-	411.0	-	406.4
2025	51,203	8,889	-	-	-	132,640	-
2026	-	-	-	-	-	204,837	409.4
2027	-	-	-	-	-	211,741	-
2028	-	-	-	-	-	218,860	411.0
2029	-	-	-	-	-	137,260	-
Total	**646,405**	**150,926**	**-**	**-**	**-**	**905,338**	**-**

Notes: Tailings quantities based on LOM plan
Placement of LOM tailings produced in November to December 2020 is considered with 2021 tailings.

18.3.4 Tailings Storage Facilities Design

18.3.4.1 North TMF Dams

The general arrangement and design configurations of the proposed North TMF ultimate dams are shown on Figure 18-4 and Figure 18-5 and summarized as follows:

- East dam ultimate crest elevation—412.6 m
- West dam ultimate crest elevation—415.0 m
- Splitter berm ultimate crest elevation—411.0 m
- Emergency spillway Invert—410.6 m.

The salient features of the North TMF east and west dams include:

- Existing dams are raised with downstream construction using sand and gravel fill (Zone 1) and/or rockfill (Zone 1A)
- Bituminous Geomembrane Liner (BGM) (Zone 5) installed on the upstream slope over bedding sand layer (Zone 1B)
- Transition layer (Zone 4) between main dam fill and bedding sand layer
- Upstream slope of 2.5 H:1V and downstream slope of 2.2 H:1V
- Emergency spillway located at east dam right abutment
- Bedrock grouting along east dam upstream plinth alignment
- Splitter berm to be constructed with end dumped rockfill (Zone 1A).

18.3.4.2 South TMF Dam

The general arrangement and design configuration of the proposed South TMF ultimate dam (north dam) are shown on Figure 18-6 and summarized below:

- North dam ultimate crest elevation—411.0 m
- Emergency spillway Invert—409.0 m.

The salient features of the South TMF north dam include:

- Downstream construction using mainly rockfill (Zone 1A)
- BGM liner (Zone 5) installed on the upstream slope over bedding sand layer (Zone 1B)
- Transition layer (Zone 4) between main dam fill and bedding sand layer
- Upstream and downstream slopes of 2.5 H:1V
- Emergency spillway located at north dam left abutment
- Bedrock grouting along north dam upstream plinth alignment

- Detailed description of North TMF and South TMF dams design and analyses and drawings can be found in the Feasibility Study Report (Wood, 2021b).

18.3.4.3 Water Balance and Hydrological Analysis

Site-wide monthly water balance was developed considering the proposed North TMF and South TMF at selected milestone years of the LOM. The following minimum pond volumes were established based on mill reclaim requirements and allowance for ice build-up during the winter months and to provide mixing water in for ETP feed.

- North TMF—100,000 m^3 from November to May, and 50,000 m^3 for the rest of the year
- South TMF—80,000 m^3 from November to May, and 50,000 m^3 for the rest of the year.

Based on the monthly water balance analyses, the required ETP capacity to manage the 1:100-year spring rain on snow event is 1,100 m^3/d for years 2021–2024. The required ETP capacity then increases from 1,100 m^3/d to 4,100 m^3/d from 2025 to 2030, mainly due to additional runoff collected from the South TMF watershed.

South TMF collects runoff from a significant watershed impacting the ETP water treatment requirement. A portion for the clean runoff reporting to the South TMF can potentially be diverted. This diversion has not been included in this feasibility design. It is recommended to assess potential clean water diversion and optimization of ETP requirement during detailed design.

18.3.4.4 Emergency Spillway Design

Emergency spillways are required for all stages of North TMF and South TMF operations. Emergency spillways for the interim stages are considered to be constructed on the dams with the final (ultimate) spillway to be located at rock abutments. Based on the flood routing analyses, the following invert elevations were established for the ultimate North TMF and South TMF Emergency Spillways:

- North TMF—El. 410.6 m at the right abutment of the east dam
- South TMF—El. 409.0 m at the left abutment of the north dam.

Each spillway is designed as a trapezoidal discharge channel with 3H:1V side-slopes, lined with Zone 5 BGM Liner and Zone 3 riprap. The discharge channels of each ultimate Emergency Spillway will be gradually fanned out (widened) to reduce water velocity at channel toe.

18.3.4.5 Liquefaction Assessment

The saturated slurry deposited tailings are generally loose to very loose and are expected to liquefy during the 1-in-2,475-year design earthquake. Filtered tailings placed at the North TMF may not liquefy for 1-in-2,475-year earthquake considering potential compaction during placement, but it is expected to liquefy (if saturated) during the 10,000-year earthquake.

Since the requirements of the *Global Industry Standard on Tailings Management* (GISTM, 2020) are being considered for seismic stability, the proposed east and west dams have been designed assuming liquefaction of both saturated slurry and filtered tailings within the North TMF.

The splitter berm is designed for 2,475-year seismic event considering that it is an internal structure contained within the perimeter dams.

Since the South TMF is contained by the single north dam, which will be built in the downstream construction method, there is no stability concern with liquefaction of tailings at that facility.

Foundation soils units at both North TMF and South TMF dams are not expected to liquefy during the 10,000-year design earthquake.

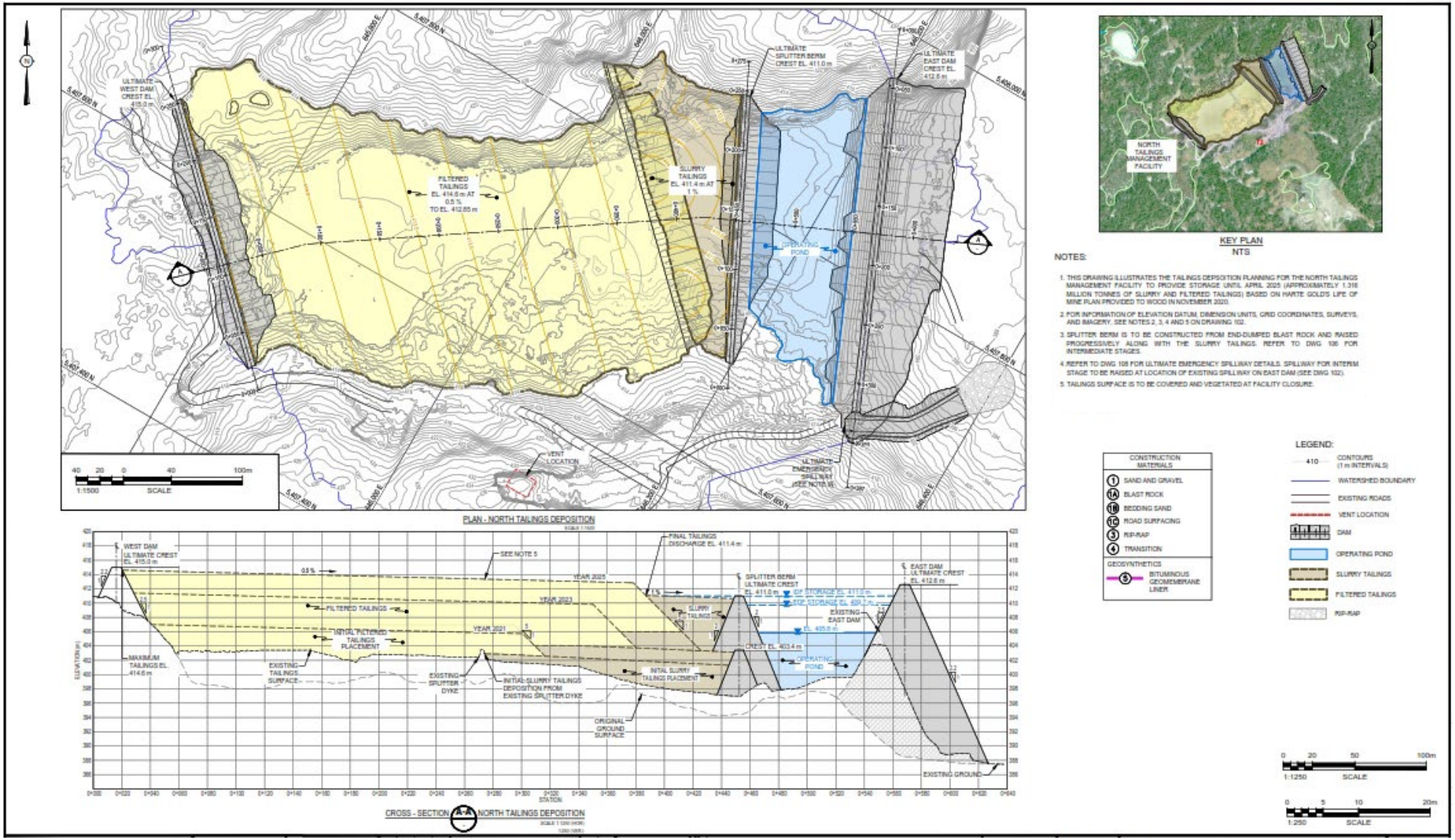


Figure 18-4: North Tailings Storage Facility—General Layout and Design Configuration (Wood, 2021b)

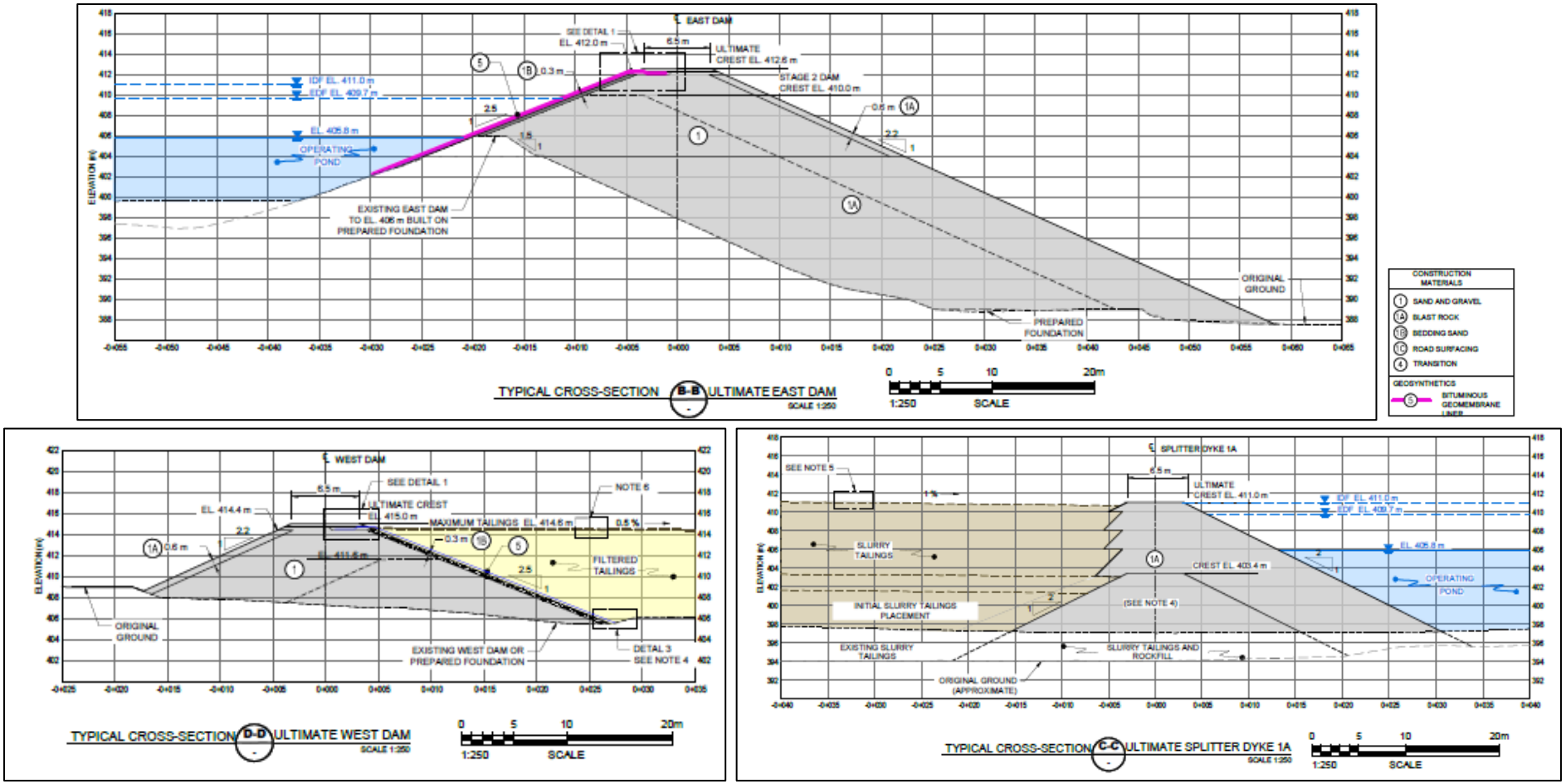


Figure 18-5: North Tailings Storage Facility—Ultimate Dam Configurations (Wood, 2021b)

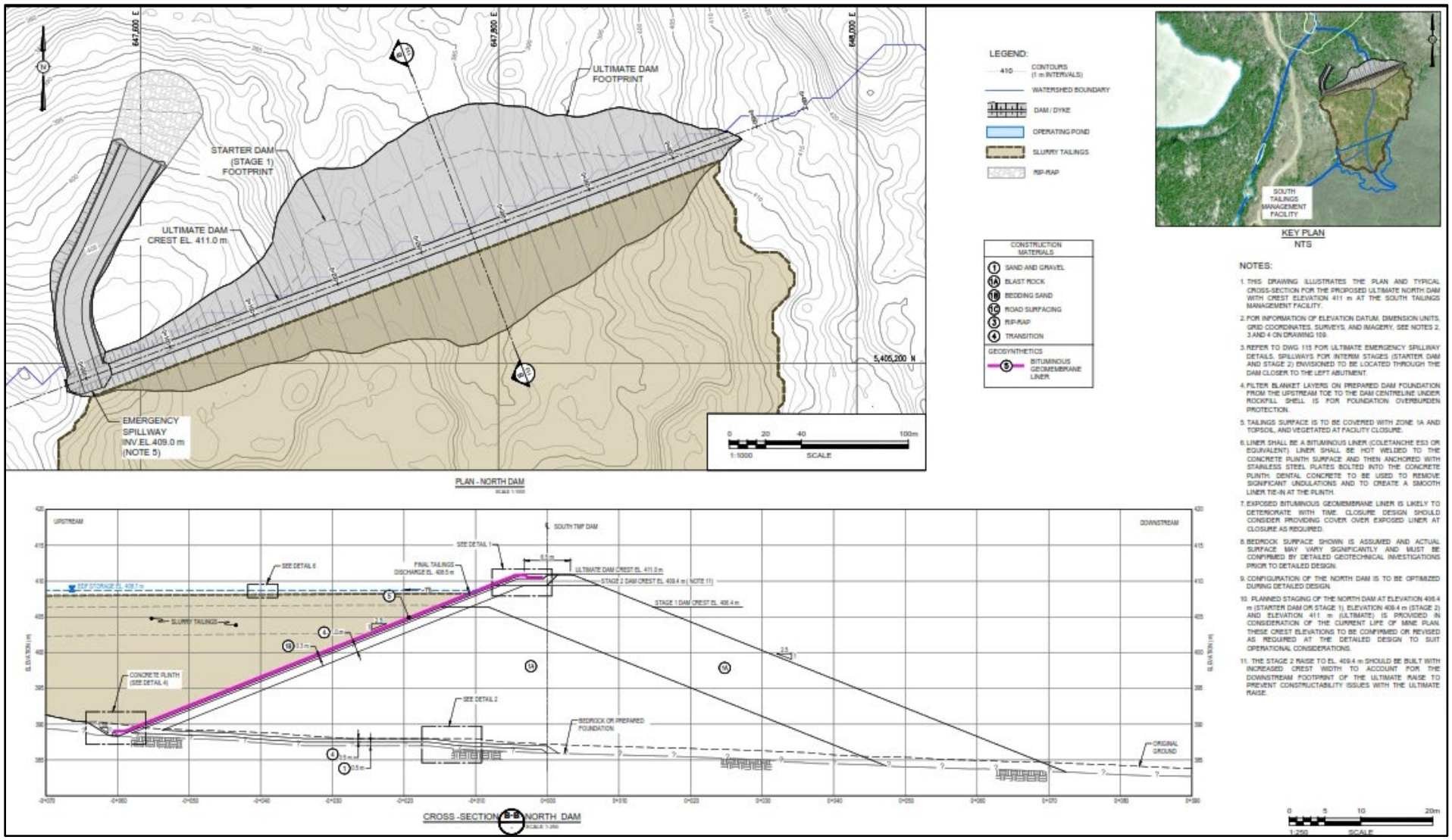


Figure 18-6: South Tailings Storage Facility—General Layout and Design Configuration (Wood, 2021b)

18.3.4.6 Slope Stability Analyses

Limit equilibrium stability analyses were performed for the proposed dam section configurations using the software Slope/W (Version 2016) and the Morgenstern-Price method of analysis. The material parameters, analysis methodologies, and results for all applicable loading conditions are presented in the feasibility study report (Wood, 2021b). Proposed ultimate dam configurations satisfy the required minimum factors of safety (FOS) for all applicable loading conditions.

Selected long-term stability analysis results for the ultimate North TMF east dam and South TMF north dam are provided on Figure 18-7 and Figure 18-8, respectively.

Internal erosion or piping are not considered to be an applicable mode of instability for the design configurations and embankment construction materials selected for the North TMF and South TMF.

18.3.4.7 Post Construction Settlement

It is anticipated that the post-construction settlement of the dams (granular earth fill) will be minimal, as the dam fill and foundation are granular, and most of the settlement is expected to occur during construction. Nevertheless, the North TMF and South TMF dams will be provided camber thickness (extra crest heights) over the design elevations to compensate for the future settlement of the dam fill and foundation soils.

To monitor crest settlement with time, survey monuments should be installed at the downstream edge of the dam crest.

18.3.4.8 Seepage Collection System

A seepage collection system should be constructed to collect runoff and seepage from the downstream slopes of the North TMF east dam and the South TMF north dam if seepage from tailings pond is observed. The collected seepage should be pumped back into the tailings ponds.

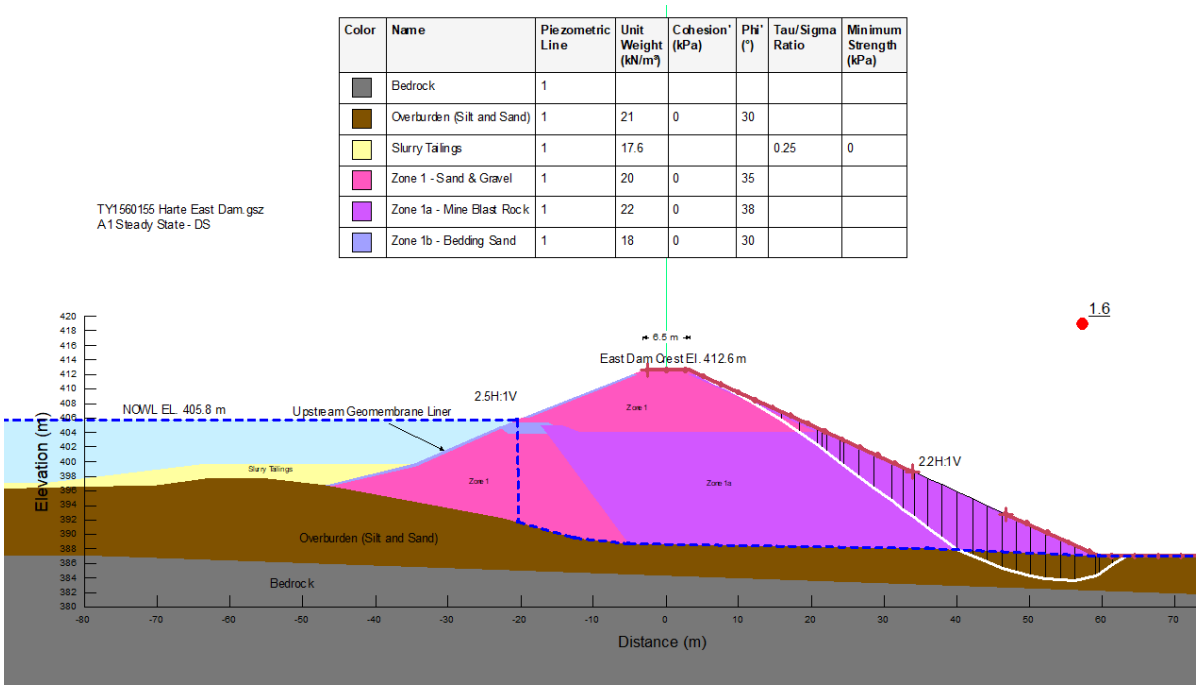


Figure 18-7: North Tailings Storage Facility—East Dam, Long-term Steady State Loading Condition (Wood, 2021b)

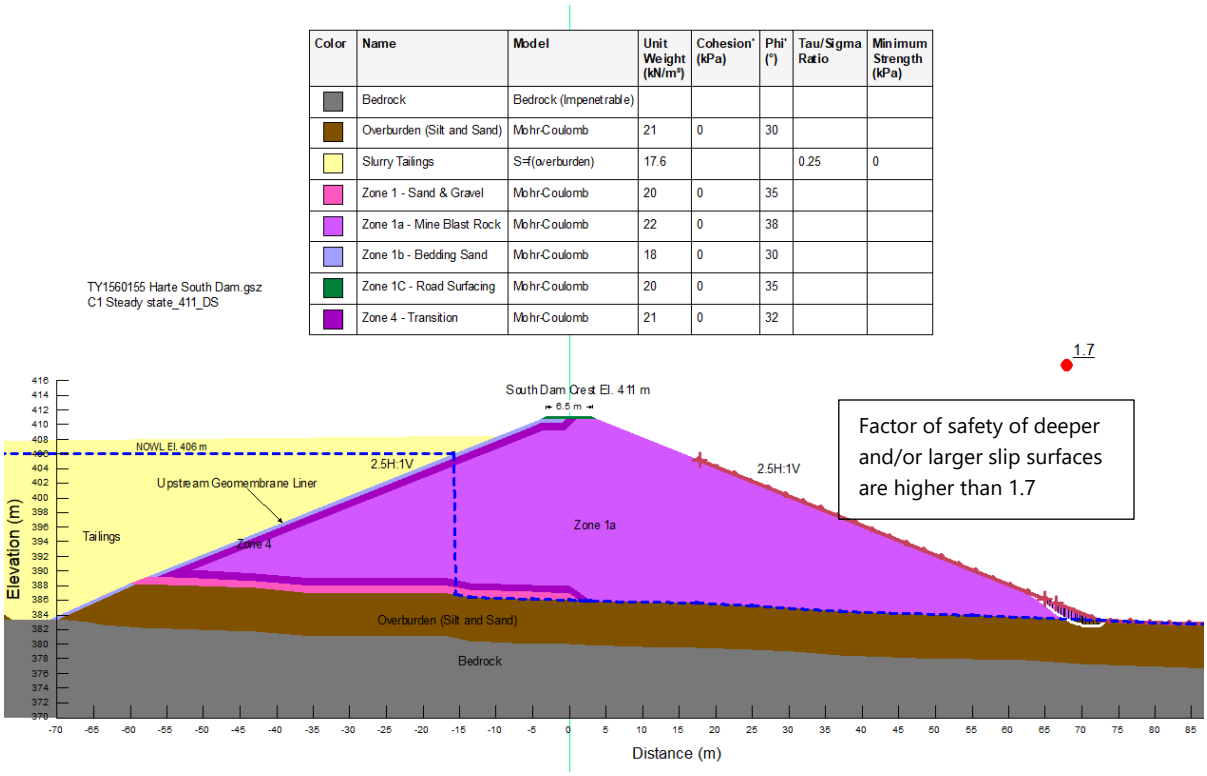


Figure 18-8: South Tailings Storage Facility—North Dam, Long-term Steady-State Loading Condition (Wood, 2021b)

18.3.5 Construction Considerations

18.3.5.1 Dam Foundation Preparation

The foundation preparation for the future North TMF dam raises and the new South TMF north dam will consist of the following:

- The footprint area of the proposed dam will be cleared of trees, bushes and brush, and the organics and loose soils will be stripped to expose competent native soil surface or bedrock.
- Proof roll the native soils and remove or replace unsuitable material, where present.
- The bedrock grouting and plinth construction along the upstream toe for the BGM liner tie-in to the bedrock will involve the following:
  - The overburden will be excavated to expose the bedrock surface along the upstream toe
  - The exposed bedrock at the BGM liner tie-in to the bedrock will be thoroughly cleaned, including open joints and fissures
  - Curtain grouted (one row directly under the liner contact point, additional rows if required)
  - Reshaped by excavation of overhangs and excessive irregularities, or dental concreting to eliminate overhangs and steep slopes or open fractures, as required

- Slush grouted in areas of minor fracturing.
- Plinth will be constructed as shown on the drawings provided with the feasibility design report (Wood, 2021b).

Grouting is not required at the North TMF west dam due to the wide filtered tailings beach and no water ponding against the dam considered in the design.

Based on the information gathered from the borehole investigations, it is expected that the curtain grouting will be required to a minimum depth of 6 m at both North TMF east dam and South TMF north dam. However, the depth and lateral extent of the grouting will be confirmed and/or modified based on grout tests to be carried out prior to and during the grouting program.

18.3.5.2 Construction Materials

The general requirements for the various fill zones to be used in the North TMF and South TMF dams are described as follows:

- Zone 1—Sand and Gravel Fill—The Zone 1 material will be used to construct most of the North TMF east and west dams. It will also be used at the base of upstream half of the South TMF north dam shell. This material shall be placed in 300 mm horizontal loose lifts and compacted to minimum 98% of the material's standard Proctor maximum dry density (SPMDD) at placement moisture content between optimum moisture content (OMC) and OMC+3%.
- Zone 1A—Blast Rock—The Zone 1A material will be used to construct the downstream section of the existing North TMF east dam raises, as required due to limited availability of Zone 1 material. The Zone 1A will also be used as main dam fill material for the South TMF. This material shall be compacted with minimum four passes of a 10-tonne smooth drum vibratory roller or uniform routing of loaded haul truck traffic. The Zone 1A will also be used to construct the splitter berm 1A by end dumping and spreading.
- Zone 4—Transition—The Zone 4 material will be used between Zone 1/1A (at the upstream slope of the dams and other areas) and the Zone 1B BGM liner bedding to provide filter compatibility between the material zones. Zone 4 material shall be placed in 300 mm loose lift and compacted at OMC ±3% to minimum 98% SPMDD using a vibratory plate compactor or by other means approved by the Owner's Representative.
- Zone 1B—Bedding Sand—The Zone 1B material will be placed as a bedding layer beneath the BGM liner. The bedding sand shall be placed in one 300 mm thick loose lifts and compacted at OMC ±3% to minimum 98% SPMDD using a vibratory plate compactor or by other means approved by the Owner's Representative.
- Bituminous Liner—The BGM Liner proposed for the dam core is Coletanche ES3 (elastometric bitumen, 4.5 mm thick) or equivalent, which is considered suitable low permeability element for cold climate applications.

18.3.5.3 Material Borrow Sources

Zone 1A Blast Rock material will be sourced from mine waste rock. Previous geochemical analysis of the mine waste rock has indicated no potential for acid drainage, and metal or arsenic leaching to the environment.

Zone 1 Sand and Gravel material is to be supplied by local contractor. There is a potential opportunity to replace some of the Zone 1 Sand and Gravel material with lower cost quarry rock in selective parts of the dam embankments. This should be considered at detailed design to optimize the cost.

It is assumed that transition material (Zone 4) will be sourced from the mine waste rock with appropriate processing.

There are thick deposits of gravelly sand and sand at some locations of the South TMF area, which may be used as Zone 1 sand and gravel or as Zone 1B bedding sand with limited processing. This should be further investigated with test pitting and laboratory tests during the detailed design.

18.3.5.4 Material Quantities and Cost Estimates

Estimated construction quantities and cost estimates over the LOM are presented in the feasibility study report (Wood, 2021b). The fill materials required for construction of the proposed dams and the general placement requirements are also described in the feasibility study report. Quantity estimates were determined along neat design lines illustrated on the accompanying design drawings.

18.4 Water Infrastructure

This section describes water-related infrastructure for the Operation site. A current water flowchart and associated water-management infrastructure are shown in Figure 18-9. Currently all water that comes into contact with the mine site is collected and treated prior to discharging into Gagegenha Lake to the east. The contact water is contained within the site watershed by natural topographical features, a cut-off wall on the west side of the portal, and by the containment pond expansion on the east side of the process plant.

The site-wide water balance currently forecasts a required discharge capacity of 2,100 m^3/d over a 200-day discharge season (mid-May to the end of November) to maintain a neutral water balance (no year-over-year accumulation) in the event of a 1:100-year spring freshet event and a 1:100-year summer EDF.

Once the South TMF is commissioned in mid-2025 all runoff that is collected in the South TMF watershed area must also be treated prior to discharge, the future water flow chart is shown in Figure 18-10. The South TMF location has a relatively large watershed and the future site wide water balance indicates that the required water discharge capacity will increase to 4,100 m^3/d (Wood, 2021b) to maintain a neutral water balance (no year-over-year accumulation) in the event of a 1:100-year spring freshet event and a 1:100-year summer EDF. Preliminary work has indicated that the

construction of a clean water diversion ditch around the South TMF could enable the discharge rate to be kept at 2,100 m^3/d, although further site investigations are required to assess the constructability and effectiveness of such a diversion ditch, therefore it has not been considered in this Feasibility Study.

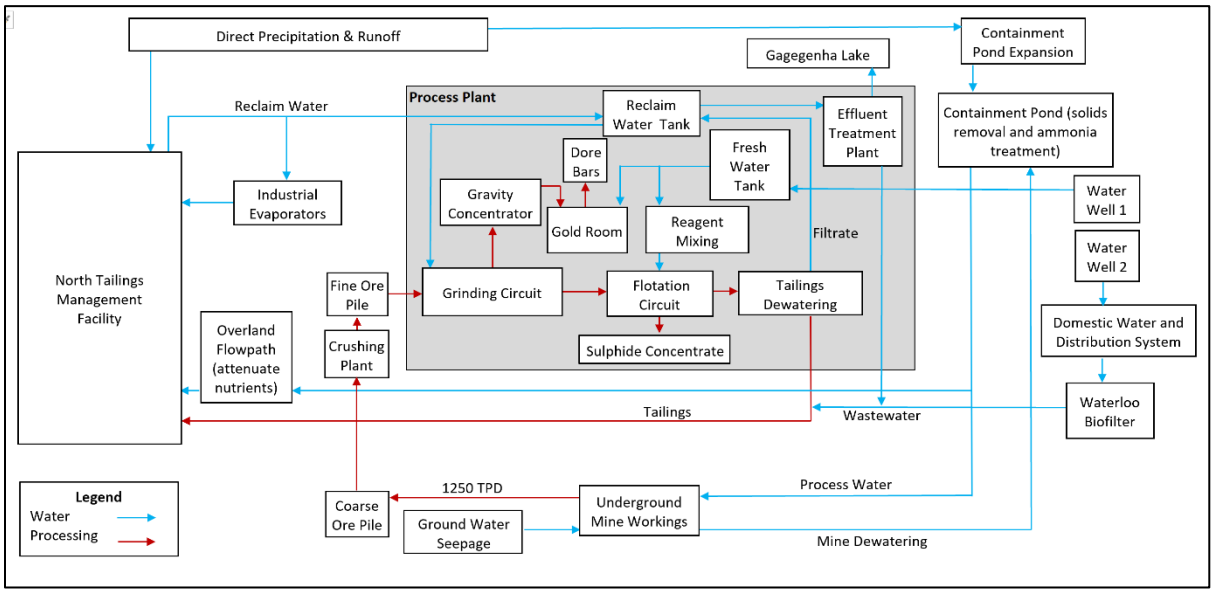


Figure 18-9: Current Water Management Flow Diagram

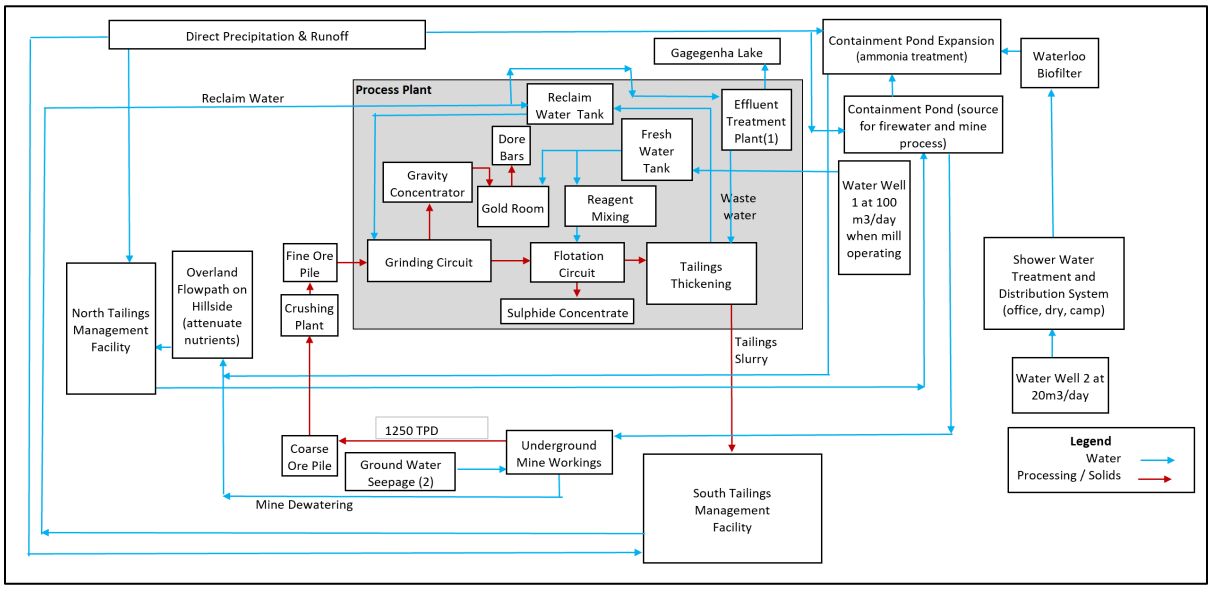


Figure 18-10: Future Water Management Flow Diagram with South TMF in Place

18.4.1 Stormwater

Stormwater reports to either the mine site containment pond or the reclaim pond of the TMF. Surplus water from the containment pond is pumped to the TMF and/or to the overland flow-path north of the mine site that drains to the TMF to attenuate any ammonia (from mine explosives) found in mine water. No untreated storm water discharge from the Operation is anticipated.

After the commissioning of the South TMF storm water will also report to the South TMF where it will be treated prior to discharge from the operation.

18.4.2 Fresh Water and Potable Water

Groundwater is currently drawn from two existing wells at the mine site, the water is considered to be non-potable and is used as service water, washing water, and for reagent mixing. Prior to usage this water is softened to remove hardness and manganese; it goes through no other water treatment process. Withdrawals from the wells for all purposes are anticipated to be below 40 m^3/d.

Potable water is provided through a commercial source and is transported to site in 20 L jugs and 0.5 L water bottles, and provided to employees and contractors free of charge.

18.4.3 Sewage Treatment

Domestic sewage is collected and treated in a containerized Waterloo Biofilter sewage disposal system located at the mine site. Treated water from the Waterloo Biofilter is pumped to the North TMF for inclusion in water recycled to the process plant.

18.4.4 Effluent Treatment

Stormwater and wastewater from the site are consolidated in the TMF. Surplus water from the TMF that is not needed for the process plant is treated and discharged. Treated effluent is conveyed to an underwater diffuser in Gagegenha Lake via a heat-traced and insulated 10 cm diameter HDPE pipeline.

The water treatment process consists of conventional chemical conditioning (pH adjustment, coagulant addition, flocculant addition), filtration to remove precipitated metals and other suspended solids, pH adjustment, and, as required, regenerable ion exchange media to remove ammonia. The turnkey design, supply, and commissioning of the water treatment plant is provided by Blumetric Environmental Inc. (Blumetric). The effluent treatment plant is housed in the process plant building and has an operating capacity ranging from 500 to 1,500 m^3/d contingent upon ammonia levels in the reclaim water.

18.4.4.1 Stage 1 Water Treatment Strategy (2021–2025)

Harte Gold was challenged in 2020 with achieving the desired discharge rates, and therefore undertook further studies with BQE Water (BQE, 2021) to assess current performance; develop an action plan and a roadmap to improving and forecasting future performance; and develop the costing around the proposed roadmap. The main contaminants or problems that have been identified as posing challenges in the treatment process and the plan to mitigate these challenges is detailed in Table 18-5.

Table 18-5: Stage 1 Water Treatment Strategy Summary

Contaminant/Problem	Description/Main Source	Mitigation Plan
Ammonia	Generated from uncombusted explosives in underground mine	2021: - Operate ETP during 'non-discharge' period to remove ammonia mass load 2022: - Increase ammonia stripping capacity by up to 300% by adding in 3 new parallel trains of ammonia stripping vessels with automation of zeolite regeneration and brine stripping
Phosphorous/Dithiophosphate (DTP)	Mill tailings (64%)—Flotation promotor Underground mine water (23%) Biofilter effluent (13%)	2021: Early 2021 testwork to confirm action plan. - Alum (PAC) addition to the existing metal precipitation reactors to enhance the removal of ortho-phosphate and particulate phosphorus in the MMF - Scavenging of residue of dithiophosphate collector by metal complexation in the tailings stream, adsorption onto resin mixed with micro-sand placed into the ballasted floc clarifier for algae removal, and/or a stand-alone GAC adsorption system used on a campaign basis if/when needed
Algae	Growth in North TMF during summer months due to nutrients and sunlight	2021: - Late 2020 upgrades to ETP included doubling of multimedia filter (MMF) capacity 2022: - Sand ballasted floc clarifier to be installed upstream of MMFs
Water Inventory	Build up of year-over-year inventory due to low discharge in 2020	2021: - Plan to purchase and install mechanical evaporators to address water inventory buildup.

The changes recommended in the water treatment flowsheet will all be able to be implemented within the current site and mill building footprint as pertinent. The ammonia stripping circuit expansion will go inside the mill building alongside the existing ETP equipment; this will expand the

total ETP footprint by approximately 70 m^2. The sand ballasted floc clarifier will only be operated seasonally and therefore can be located outside of the mill building in a location that enables simple incorporation into the flowsheet. Mechanical evaporators will be placed around the North TMF in locations that consider the prevailing wind directions to ensure that undissipated droplets do not settle outside of the mine site's watershed.

18.4.4.2 Stage 2 Water Treatment Strategy (2025–2029)

The Stage 2 water treatment strategy (Stage 2) is characterized by the incorporation of the South TMF to the site-wide water balance. The large increase in catchment area by the addition of the South TMF is the primary cause for the increase in required discharge capacity. As the rate of tailings production and mine production (ammonia loading) does not change with the commissioning of the South TMF, the fresh water runoff collected in the South TMF basin will serve to dilute existing contaminant levels and improve the overall water quality on site.

During Stage 2 Harte Gold will be depositing slurry tailings in the South TMF through a heat-traced slurry pipeline and all mill reclaim water will also be drawn from the South TMF through a heat-traced reclaim water pipeline. During Stage 2 all mine water and treated sewage water (grey water) will continue to report to the North TMF. This water flowchart ensures that the quality of water in the South TMF during the discharge season will be high, and minimal treatment of this water will be required. An additional water treatment plant for South TMF will be required, and is anticipated to be a containerized treatment plant that sits in proximity to the process plant complex. The treatment plant for the South TMF water will consist of:

- Reagent injection and reaction tanks for metals removal by precipitation
- Multi-media filtration for removal of metal precipitate and suspended solids.

This strategy is beneficial to reaching the required discharge rate as the treated South TMF water can be used to dilute the contaminated North TMF water to meet effluent discharge limits. The Stage 1 upgrades to the existing ETP will also ensure that the ETP is capable of meeting the ammonia mass removal requirements.

One important aspect to consider is that during the commissioning of the South TMF some water will need to be directed to the South TMF to partially pre-fill or "charge" the new dam prior to tailings deposition to facilitate water reclaim. As it is important to keep the new facility largely free of ammonia and chloride, there should be special procedures followed for this pre-filling process. During the transition period, the last year when tailings are deposited in the North TMF all underground mine water and contact water, to the extent possible, should bypass the North TMF and report directly to the ETP, and the treated effluent and any runoff collected in the South TMG catchment should be used to pre-fill the South TMF.

18.5 Waste Rock Dump

Mine waste rock from underground is currently being placed and managed at the area between the processing plant and the existing North TMF. It is expected that majority of the mine waste rock will be used for progressive dam construction at both the South and North TMFs. Over the LOM just over 2 Mt of waste rock are expected to be hauled to surface from the underground workings, and the planned TMF construction and closure activities are expected to consume up to 1.5 Mt.

Surplus waste rock will be placed either within the mine site watershed or the TMF watershed, south of the North TMF. Waste rock lifts will be placed with stable embankment slopes adequate for long-term physical stability so that no further work is required at closure. Runoff water from these locations is fully contained.

In 2016, Amec Foster Wheeler (now Wood), evaluated the geochemical properties of waste rock, mineralized material, and tailings (Amec Foster Wheeler, 2016c & 2016d). Based on static testing results it was determined that these materials present a low risk of acid generation and metal leaching. Supplemental kinetic tests are currently continuing to characterize the acid generation potential of an isolated zone of low sulphide, low carbonate waste rock. This isolated zone had indicated an uncertain acid generation potential based on static testing. If the rock at the isolated zone is determined to be potentially acid generating and is encountered during mining activities, it will be used as mine backfill, or crushed and used as underground road ballast.

19 MARKET STUDIES AND CONTRACTS

No market studies have been conducted by Harte Gold. Gold is a freely traded commodity on the world market for which there is a steady demand from numerous buyers.

Harte Gold presently produces gold in both gold doré bullion and gold flotation concentrates.

Gold doré bars are presently sent to Asahi Refining Canada Ltd. (Asahi) located in Brampton, Ontario for refining. Transportation and refining charges have been agreed to and are typical of charges in the industry.

In respect of refined gold produced from doré bars, Harte Gold has three off-take agreements pursuant to which Harte Gold is obliged to deliver a portion of the resultant ounces until the total number of ounces required to be delivered have been delivered. In particular:

- Orion Mine Finance (Orion) is entitled to 50% of the refined doré ounces until 375,000 oz have been delivered.
- ANR Investments B.V. (Appian) is entitled to (i) 11.5% of the refined doré ounces until 93,750 oz have been received and (ii) 18.5% until 500,000 of the refined doré ounces have been delivered.

Each of Orion and Appian then pay Harte Gold for such ounces based on the gold price quoted by the London Bullion Metals Association (LBMA).

Harte Gold has entered into a contract with Glencore Canada Corporation (Glencore) to sell gold concentrates to Glencore's Horne Smelter in Rouyn, Quebec. The term of the contract runs to 2030, cancellable any time after February 2020 with six months' notice (Glencore, 2020). Glencore purchases the gold concentrates on arrival at the Horne Smelter at the market gold price less treatment and refining charges. The gold price is based on the gold price quoted by the LBMA.

Harte Gold has entered into an IIBA with PMFN, the proximal First Nation, in connection with the Company's Sugar Zone Property. The IBA applies to all mines that may be developed on the 81,287 ha Sugar Zone Property and provides the framework within which Harte Gold and PMFN will continue to work together during the production phase of the Sugar Zone Mine. The key IBA financial terms include a 4% net profits interest (NPI), based on the World Gold Council definition of "all-in sustaining cost" metrics.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Sugar Zone gold property is in an area of northern Ontario that has hosted mineral exploration and mining projects for more than 50 years. The Property has a history of gold exploration that began in 1969. Previous exploration activities on the Property have comprised surface drilling, geophysical surveys, geological mapping, and exploration trail development. No environmental liabilities exist related to previous exploration activities.

20.1 Environmental Regulatory Setting

The environmental assessment (EA) and permitting framework for metal mining in Canada is well established. Following successful completion of an EA process, a project undergoes a permitting phase to allow operations to proceed. The Project is then regulated through all phases (construction, operation, closure, and post-closure) by both federal and provincial departments and agencies.

20.1.1 Environmental Assessment Process

The federal impact assessment and provincial EA processes provide mechanisms for reviewing major projects to assess potential impacts. This legislation promotes responsible environmental decision-making and ensures that stake holders and interested parties have an opportunity to comment on proposed projects.

20.1.1.1 Federal Impact Assessment Requirements

The Project is not required to complete a federal impact assessment because the production capacity increase associated with the Project is below the threshold stated in Sections 19(c) and 19(d) of the *Physical Activities Regulations* made under the *Impact Assessment Act*, as summarized below:

- Maximum ore production capacity after the expansion would be below 5,000 t/d
- Maximum ore input capacity to the mill after the expansion would be below 5,000 t/d.

However, under Section 9(1) of the *Impact Assessment Act*, the Minister of Environment and Climate Change may designate a project that is not prescribed by the Physical Activities Regulations if adverse effects or public concerns warrant a federal impact assessment. Harte Gold has implemented proactive environmental practices to prevent adverse effects and public concerns, effectively minimizing the likelihood of a designation by the Minister to make the Project subject to a federal impact assessment.

20.1.1.2 Provincial EA Requirements

Harte Gold has completed the provincial class EAs listed below:

- Class EA for Resource Stewardship and Facility Development, in accordance with MNR (2003)
- Class EA for <5 MW Cumulative Capacity of Diesel Generators, in accordance with Ontario Regulation 116/01.

The Project is not required to complete any additional provincial EA processes. However, interested parties may make a designation request to the Ontario Ministry of the Environment, Conservation and Parks (MECP) Minister to have a project referred to an individual EA. MECP assesses the merits of the request, and the Minister makes a decision whether or not to grant the request. The above-noted Class EA processes and numerous permit application postings on Ontario's Environmental Registry (http://www.ebr.gov.on.ca) have not resulted in designation requests for an individual EA.

Legislative changes to the provincial EA process have been proposed by the current government and will need to be confirmed prior to Project implementation.

20.1.2 Current Regulatory Status

Approvals currently in force for the Project are presented in Table 20-1.

Mining leases (surface and mining rights, pursuant to Section 81 *Mining Act*) 109593, 109592, 109602, and 109605 are in place for the mine site and surrounding area, providing the ability for Harte Gold to clear land and construct Project components such as roads and trails. The mining leases adjoin the all-weather road network of the White River Forest SFL that provides year-round access.

Permits or permit amendments that would be required to implement the Project are listed in Table 20-2.

Table 20-1: Current Approvals

Permit[1]	Regulatory Agency	Relevant Legislation	Year of Issuance / Last Amendment	Rationale
Permit to Take Water 8337-B3QJAH	Ministry of Environment Conservation and Parks	Ontario *Water Resources Act*	2018	Withdrawal of water from underground workings
Permit to Take Water P-300-6062209462			2019	Withdrawal of surface water from Gagegenha Lake, Dayohessarah Lake, and unnamed pond
Permit to Take Water 5567-BVBPWM			2020	Withdrawal of groundwater from water supply wells at Sugar Zone Mine
Sewage Environmental Compliance Approval 8479-B3QHJZ		*Environmental Protection Act*	2018	Approve industrial and domestic sewage works
Air Environmental Compliance Approval 7924-BBLL6K			2019	Approve air emissions from site
Land Use Permit 1524-1001682	Ministry of Natural Resources and Forestry	*Public Lands Act*	2018	Approve use of Crown-owned surface rights for power line
Land Use Permit 1526-1001416			2013	Approve use of Crown-owned surface rights along Gagegenha Lake shoreline
LRIA Approval No. WAWA-LOC-01-2017		*Lakes and Rivers Improvement Act*	2017	Location Approval of North TMF (plans and specifications approval for this offline structure issued by MENDM as part of 2018 Closure Plan Amendment)
LRIA Approval No. WAWA-LOC-01-2020			2020	Location Approval of South TMF
Sugar Zone Project Closure Plan (as amended)	Ministry of Energy, Northern Development and Mines	*Mining Act*	2020	Approve development and closure of the Sugar Zone Mine
Encroachment Permit EC-2018-50S-00000030 V1	Ministry of Transportation	*Public Transportation and Highway Improvement Act*	2018	Construction and operation of power line within Highway 631 corridor

Note: [1] Excludes Forest Resource Licenses for harvesting Crown-owned merchantable timber.

Table 20-2: Permits Required to Implement the Project

Permit / Permit Amendment	Regulatory Agency	Relevant Legislation	Rationale for Permit Issuance	Rationale for Amendment
Amendment to Sewage Environmental Compliance Approval 8479-B3QHJZ	Ministry of Environment, Conservation and Parks	*Environmental Protection Act*	Approves industrial and domestic sewage works	Upgrade to ETP and increased effluent discharge volume, development of South TMF
Amendment to Air Environmental Compliance Approval 7924-BBLL6K		*Environmental Protection Act*	Approve site wide air emissions	Increased production rate and associated modifications to air emission sources
Amendment to Sugar Zone Project Closure Plan	Ministry of Energy, Northern Development and Mines	*Mining Act*	Approve development and closure of the production of the Project.	Increased production rate; changes to site footprint including development of the South TMF and additional financial assurance
Plans and Specifications Approval for Stage 1 South TMF	Ministry of Natural Resources and Forestry	*Lakes and Rivers Improvement Act*	Approves development of South TMF	Approves construction of the Stage 1 South TMF; approvals for subsequent stages to be obtained on an as-needed basis
Forest Resource License		*Crown Forest Sustainability Act*	Permit not yet issued	Approve the harvest of Crown owned merchantable timber in the area of the South TMF

20.2 Environmental Studies and Management

This section indicates the environmental studies that have been completed for the Project, environmental aspects of concern, and current management systems that are in place for the Project.

20.2.1 Environmental Studies

The Sugar Zone Mine closure plan describes current conditions at the Property. Baseline monitoring activities and areas of study to date are listed below and have been incorporated into the closure plan, annual environmental performance reports, and other submissions to regulatory agencies:

- Archaeological assessment
- Species at risk and significant wildlife habitat surveys since 2009
- Aquatic biological assessment and delineation of fish habitat
- Effluent mixing and plume-delineation studies
- Hydrogeological characterization
- Geochemical characterization of development rock, mineralized material, and tailings
- Geotechnical assessments of underground workings to ensure a stable crown pillar
- Semi-annual sampling of groundwater monitoring wells since 2016
- Regular surface-water monitoring since 2009 in the vicinity of the Project site.

No biological values (e.g., species at risk, ecologically significant features, regionally significant wetlands, significant wildlife habitat) or environmentally sensitive areas that would preclude the

development of the Project site have been identified to date. Archaeological assessment work by Horizon Archaeology and Northwest Archaeological Assessments, with participation of technicians from the PMFN, did not identify any areas with a high potential to host cultural heritage values at the Project site.

20.2.2 Environmental Management

Harte Gold has developed an environmental management system (EMS) for the Project. The EMS is a plain-language tool to identify and help manage environmental compliance obligations for the Property. The extent of the Property covered by the EMS includes the Project site, as well as assets in the town of White River and the access corridor. The elements of the EMS are:

- Lists of the relevant legislation, approvals, agreements, and documents that contain Harte Gold's environmental obligations
- Property is divided into discrete environmental management areas; each area has a description of the environmental obligations, inspection items, and the corresponding inspection frequency
- Designated inspectors and documented inspection protocols
- Procedures to deal with real and potential non-compliance issues
- Guidance for documentation requirements, regular updates, and regular internal reporting on performance and auditing.

The EMS identifies the Project's compliance obligations, and outlines inspection/audit protocols to ensure compliance issues are identified, documented, reported, and mitigated. The EMS also addresses community engagement/consultation obligations, and includes a commitments registry. The EMS will evolve into a tool to manage corporate social responsibility commitments and obligations.

Monitoring of surface-water and groundwater quality surrounding the Project will continue for the life of the Project and also post closure.

20.2.3 Environmental Aspects and Sensitivities

The Project site adjoins the Dayohessarah Lake watershed; the lake is valued for recreation. As such, emphasis has been placed on controlling discharges of water, fugitive dust, and noise.

20.2.4 Waste Rock

Waste rock will be used for construction purposes solely at the mine site and for TMF construction. Surplus waste rock will be placed either within the mine-site watershed or the TMF watershed. Waste-rock lifts will be placed with stable embankment slopes that are adequate for long-term physical stability so no further work will be required at closure.

Amec Foster Wheeler was engaged to evaluate geochemical properties of waste rock, mineralized material, and tailings. Amec Foster Wheeler (2016) characterizes the Project as generally having a low risk of acid generation and metal leaching. Supplemental kinetic testing is continuing at the present time. If any development rock is determined to be potentially acid-generating at a future date, it will be used as mine backfill below the post-closure water level in the flooded mine. Harte Gold has implemented a proactive monitoring plan for the life of the mine to identify potential chemical stability concerns.

20.2.5 Water Discharge

Water discharge will be minimized by recycling water to the extent practical using the following practices:

- Mine water will continue to be recycled from underground sumps or from the containment pond. Fresh water will not be pumped into the mine.
- Water will be accumulated in the TMF and recycled to the process plant for use in the process.
- Runoff from the mine site ultimately reports to the North TMF; there is one discharge point for the Project that uses an underwater diffuser in Gagegenha Lake.

Surplus water from the TMFs that is not needed for processing will be treated and discharged to the underwater diffuser in Gagegenha Lake that has been engineered for efficient mixing. The water-treatment process consists of conventional chemical conditioning (pH adjustment, coagulant addition, flocculant addition, metal precipitant addition); filtration to remove precipitated metals, phosphates, and other suspended solids; regenerable ion exchange media to remove ammonia; and final pH adjustment. Reverse osmosis treatment is a short-term contingency during early spring discharge to remove residual mill reagents that may not have sufficiently degraded to their environmentally benign components over the winter months.

Harte Gold began an assimilative capacity study in 2016, with input from MECP, to derive receiver-based effluent criteria in accordance with MECP policy outlined in MOE (1994a) and MOE (1994b). There were no impacts observed from local mineralization; the effluent criteria are presented in Table 20-3.

Operational strategies that will be employed to help meet effluent criteria include good blasting practices in the mine, to minimize ammonia and aeration of the containment pond and TMF reclaim ponds to prevent elevated nitrite and solubilization of phosphorus from sediment under anoxic conditions.

The Project will follow a conventional seasonal discharge window from spring melt until freeze-up pending further consultation with MECP.

The capacity of the ETP combined with the storage in the TMFs is adequate to manage the 1-in-a-100-year environmental design flood.

Table 20-3: Effluent Criteria

Parameter	Receiver Based Effluent Criteria	
	Daily Concentration	Monthly Average
TSS (mg/L)	30	15
Total Ammonia Nitrogen (mg/L)	10	5
pH Range (Units)	6.5 to 8.5	
Total Arsenic (mg/L)	0.1	0.05
Total Boron (mg/L)	5	2.5
Total Cobalt (mg/L)	0.014	0.007
Total Copper (mg/L)	0.1	0.05
Total Lead (mg/L)	0.08	0.04
Total Nickel (mg/L)	0.6	0.3
Total Zinc (mg/L)	0.4	0.2
Total Phosphorus	0.26	0.13
Rainbow Trout and Daphnia Magna (Acute Lethality)	Less than 50% mortality in 100% effluent	

Note: MECP may revise site-specific effluent criteria as part of the application to amend the sewage Environmental Compliance Approval (refer to Table 20-2).

20.2.6 Fugitive Dust

Air emission sources will comprise diesel-fired equipment, diesel generators, propane- and natural-gas-fired combustion heating units, return air from the underground workings, as well as fugitive dust emissions from vehicle operation, the TMFs, crushing, and on-surface material handling typically associated with an underground mining and milling operation. Practices to minimize fugitive dust are listed below:

- Minimize vehicle speed and travel time, use dust suppressants on travelled roads, minimize track-out of fines from material handling areas
- Minimize coarse mineralized-material stockpile size, enclose the fine-mineralized-material stockpile, and use buildings and treelines as windbreaks to the maximum extent practical
- Frequent relocation of the tailings discharge to maintain a wetted tailings surface
- Tackifier could be added to deposited tailings to bind together the tailings solids and prevent entrainment by wind
- Enclose material transfer points and use water sprays to suppress dust
- Other applicable best practices listed in Environment Canada (2009).

Harte Gold has developed, and implements, a best management practices plan for controlling fugitive dust. Fugitive dust is considered in the site-wide emission summary and dispersion modelling (ESDM) report that is prepared to support the Air ECA amendment application listed in Table 20-2. The ESDM report demonstrates compliance with MECP air quality criteria during worst-case scenarios.

20.2.7 Noise

A remote tourism lodge on the west shore of Dayohessarah Lake is approximately 3.5 km from the underground portal. Harte Gold is designing infrastructure for the Project so that noise emissions from the site will be largely controlled to protect the lodge. Modern noise-abatement measures have been integrated into the Project design, such as placing enclosures around noise sources, strategically situating building openings, and installing mine fans below grade.

20.3 Social and Community

The area is characterized by wilderness, forestry, and mineral exploration land uses. The Property is within the White River Forest Sustainable Forest License.

There are no known abandoned mine hazards within the area of the Sugar Zone Project closure plan, and all known mine hazards are addressed in the closure plan. A review of the ENDM's Abandoned Mine Information System did not identify mine hazards in the vicinity of the Project.

The nearest private interest in the region is a tourism operator with camps on Dayohessarah Lake and on Hambleton Lake. Harte Gold maintains a positive working relationship with this tourism operator.

20.3.1 Indigenous Consultation

Harte Gold has undertaken consultation on behalf of the Crown, and as directed by Ontario Ministry of Energy, Northern Development and Mines (MENDM), regarding exploration and development activities on the Property since June 2010. The Crown has delegated procedural aspects of consultation to Harte Gold for the PMFN and the Biigtigong Nishnaabeg. Harte Gold supports the Crown in consultation with other First Nations, as the Crown requests.

Consultation to date with First Nations has not identified the presence of cultural heritage values that would be affected by the Project.

20.3.2 Pic Mobert First Nation

Harte Gold entered into an agreement with the PMFN in 2018. The agreement provides a framework within which Harte Gold and the PMFN will continue to work together during the production phase of the Sugar Zone Mine.

20.3.3 Biigtigong Nishnaabeg (Ojibways of the Pic River Heron Bay First Nation)

Harte Gold entered into an agreement with Biigtigong Nishnaabeg in 2018. The agreement provides a framework within which Harte Gold and Biigtigong Nishnaabeg will continue to work together during the production phase of the Sugar Zone Mine.

20.3.4 Public Consultation

Consultation with the local and regional communities has commenced and will continue as the Project evolves. This will include meeting with the municipal and provincial government as well as other parties. This consultation will include meetings, public information sessions, and other communications to ensure stakeholders are aware of Harte Gold's plans, and concerns can be identified and resolved in an efficient manner. Table 20-4 summarizes formal public consultation to date.

Table 20-4: Public Consultation to Date

Date	Summary of Public Consultation Undertaken	Summary of Information Provided[1]	Summary of Comments Received
2012	Public information session in White River in December, in accordance with Section 140 *Mining Act* and Section 8 Ontario Regulation 240/00; in accordance with MNRF requirements for a Category B Class EA. This session was also held as part of the Class Environmental Assessment process required pursuant to Ontario Regulation 116/01 for the installation of diesel generators	Published newspaper article; mailed notices to regional stakeholders; posted notices at White River post office and Royal Canadian Legion; notified relevant government agencies Overview PowerPoint presentation of Sugar Zone Project, property history, and every general aspect of the Advanced Exploration phase of the Project	No comments or concerns received in relation to any aspect of the Project. Inquiries regarding employment and contracting opportunities
2016	Public information session in White River in November, in accordance with Section 141 *Mining Act* and Section 8 of Ontario Regulation 240/00.	Published newspaper article; posted notices at post office and Royal Canadian Legion; notified relevant government agencies Overview PowerPoint presentation of Sugar Zone Project, property history, and every general aspect of Phase I production	No comments or concerns received in relation to any aspect of the Project. Inquiries regarding employment and contracting opportunities
2017	Project update session in White River in April, in accordance with procedure in Section 8 of Ontario Regulation 240/00	Published newspaper article; posted notices at White River and Hornpayne post offices and Royal Canadian Legion in White River; notified relevant government agencies Overview PowerPoint presentation of Sugar Zone Project, property history, and every general aspect of commercial production	No comments or concerns received in relation to any aspect of the Project Inquiries regarding employment and contracting opportunities
2012 to 2018	Posted applications on the Environmental Bill of Rights for Project approvals and amendments		No comments received

Note: Newspaper articles are placed in *Marathon Mercury and Algoma News*.

20.4 Closure

For a project to proceed, a closure plan must be filed that meets requirements of the *Mine Rehabilitation Code of Ontario* contained in Ontario Regulation 240/00 (*Mining Act*) and is consistent with any traditional land uses and occupancy by Indigenous communities. A closure plan outlines how project lands will be rehabilitated to a productive land use post closure, meets the requirements of

the *Mine Rehabilitation Code of Ontario,* and describes the costs associated with doing so, as well as implements a monitoring program. Based on currently available information, there are no chemical stability concerns associated with rock or tailings; thus, the rehabilitation measures will focus on physical stability and compliance with the *Mine Rehabilitation Code of Ontario*.

To ensure that the rehabilitation work outlined in a closure plan is successfully performed, financial assurance equal to the estimated cost of the rehabilitation work must be provided by the proponent, to be held in trust by the MENDM. Financial assurance must be included with the submission of a closure plan.

General elements of the closure plan for the Project are summarized below:

- Buildings, infrastructure, and equipment that are not required for water management will be removed and salvaged, recycled, or disposed of. Contractor-owned items and leased items will be removed by their respective owners.
- Contaminated soil will be managed in accordance with MECP requirements.
- The ventilation raises will be sealed using concrete caps, and the portal will be barricaded. A bulkhead will be installed in the drift that connects the Sugar Zone to the Middle Zone to prevent water decanting out of the vent raises at the Middle Zone when the mine is flooded at closure.
- The mine site will be scarified, and fill embankments will be sloped for long-term physical stability. Soil from a local-soil stockpile will be placed over the mine site, and the area will be revegetated using a commercially available seed mix prior to planting seedlings consistent with prescriptions in the White River Forest Management Plan.
- The TMFs will be dewatered and covered with a lift of soil that is tilled and seeded. There are no chemical stability concerns, so an engineered cover is not required.
- The TMF reclaim ponds and mine site containment pond will be treated and discharged to the underground mine to accelerate flooding. Once the incoming runoff to these areas meets the water-quality criteria that are developed with MECP and MENDM, the spillways will be modified to allow flow-through.
- The electricity distribution system and water-management infrastructure will finally be removed once active water management is no longer required.
- Roads on the Property will be rehabilitated as they are removed from use, in general accordance with MNR (1990).
- Post-closure monitoring will carry on for 10 years.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

The capital cost estimate was prepared by Harte Gold with input, in key areas, from the following companies:

- Process plant expansion—Halyard
- Paste backfill plant upgrade—P&C
- Paste backfill underground distribution system—P&C
- Tailings storage facilities—Wood
- Mining unit rates—Stantec
- Underground electrical distribution, pumping, and water supply—Stantec
- Water treatment—BQE
- Mine ventilation—JMVL.

Table 21-1 provides a summary of the LOM capital cost. Capital is categorized as sustaining if it is required to maintain the current operation at a production rate of 800 t/d. Expansion capital relates to projects that are required in order to achieve the 1,200 t/d throughput rate.

All costs are presented in Canadian dollars ($) and are based on Q4 2020 pricing. The estimates are deemed to have an accuracy of ±15%, in line with accepted feasibility study standards.

Table 21-1: Capital Cost Summary

Sustaining Capital		Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030-39
Mining	Development	$ '000s	**199,229**	25,602	22,081	21,333	24,105	22,785	23,675	24,346	22,275	13,026	-
	Infrastructure	$ '000s	**39,651**	6,339	10,300	4,126	4,111	3,877	4,012	2,849	2,239	1,798	-
	Equipment Overhaul/Repl.	$ '000s	**18,209**	902	2,594	2,806	1,804	3,758	2,772	2,273	1,298	-	-
Processing		$ '000s	**765**	765	-	-	-	-	-	-	-	-	-
Surface Infrastructure		$ '000s	**3,841**	2,541	1,300	-	-	-	-	-	-	-	-
Tailings		$ '000s	**27,465**	4,334	2,552	5,922	7,603	3,090	2,167	150	1,646	-	-
Paste Backfill		$ '000s	**26,121**	2,819	10,931	3,237	1,691	1,684	1,790	1,620	1,433	916	
Project Management		$ '000s	**1,278**	426	426	426	-	-	-	-	-	-	-
Closure		$ '000s	**8,177**	-	-	-	-	-	-	-	-	-	8,177
Grand Total		**$ '000s**	**324,736**	**43,729**	**50,184**	**37,850**	**39,315**	**35,194**	**34,416**	**31,239**	**28,892**	**15,740**	**8,177**

Expansion Capital	Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Mining Equipment	$ '000s	**4,615**	-	2,307	2,307	-	-	-	-	-	-
Process Plant	$ '000s	**16,347**	2,381	13,965	-	-	-	-	-	-	-
Permitting	$ '000s	**136**	136	-	-	-	-	-	-	-	-
Grand Total	**$ '000s**	**21,097**	**2,517**	**16,273**	**2,307**	-	-	-	-	-	-

21.1.1 Mine Capital Cost

Table 21-2 shows the mining sustaining capital estimate. All underground development is categorized as sustaining capital because it is required whether or not the production rate is increased. The average unit cost for capital development is $5,750/m. The remainder of the sustaining capital relates to major underground infrastructure and additional equipment required to cater for increased haulage distances as operations progress deeper.

Table 21-2: Mining Sustaining Capital Estimate

		Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Development	Lateral	$ '000s	84,401	12,322	9,694	9,220	10,188	9,600	9,724	10,355	8,796	4,504
	Vertical	$ '000s	6,160	199	707	561	748	464	924	1,105	934	520
	Haulage	$ '000s	22,392	1,696	1,631	2,302	2,726	2,929	3,224	3,200	3,191	1,493
	Indirects	$ '000s	86,275	11,386	10,049	9,249	10,444	9,792	9,804	9,686	9,354	6,510
	Total	**$ '000s**	**199,229**	**25,602**	**22,081**	**21,333**	**24,105**	**22,785**	**23,675**	**24,346**	**22,275**	**13,026**
Infrastructure	UG Electrical	$ '000s	14,319	2,400	2,339	1,902	1,317	1,317	1,317	1,243	1,243	1,243
	Ventilation Upgrades	$ '000s	5,445	-	5,445	-	-	-	-	-	-	-
	Vent Walls & Doors	$ '000s	4,685	225	580	670	670	670	670	540	420	240
	Refuge Stations	$ '000s	3,410	440	440	440	440	440	440	440	220	110
	Pump Stations	$ '000s	2,343	599	338	-	938	469	-	-	-	-
	Mid Zone RAR	$ '000s	2,107	2,107	-	-	-	-	-	-	-	-
	UG Workshop	$ '000s	1,888	-	472	472	-	-	944	-	-	-
	ESS Construction	$ '000s	1,613	224	224	179	179	179	179	179	179	90
	Egress Raises	$ '000s	1,336	164	164	164	164	164	164	164	123	62
	Sump Pumps	$ '000s	864	108	108	108	108	108	108	108	54	54
	Explosives Magazine	$ '000s	560	-	93	93	93	93	93	93	-	-
	UG Booster Fans	$ '000s	445	-	-	-	105	340	-	-	-	-
	Auxiliary Fans	$ '000s	327	72	45	45	45	45	45	30	-	-
	Cap Magazine	$ '000s	309	-	51	51	51	51	51	51	-	-
	Total	**$ '000s**	**39,651**	**6,339**	**10,300**	**4,126**	**4,111**	**3,877**	**4,012**	**2,849**	**2,239**	**1,798**
Equipment	Additional Equipment	$ '000s	4,880	573	1,796	-	1,256	1,256	-	-	-	-
	Major Overhaul	$ '000s	13,330	330	798	2,806	549	2,503	2,772	2,273	1,298	-
	Total	**$ '000s**	**18,209**	**902**	**2,594**	**2,806**	**1,804**	**3,758**	**2,772**	**2,273**	**1,298**	**-**
Grand Total		**$ '000s**	**257,089**	**32,844**	**34,975**	**28,265**	**30,020**	**30,420**	**30,460**	**29,469**	**25,812**	**14,824**

Table 21-3 shows the underground expansion capital required to increase production to 1,200 t/d. In this study the equipment has been specified as direct purchase by Harte Gold; however, the option remains open to lease this equipment which would delay the initial capital outlay. Quotes for this equipment were received from Original Equipment Manufacturers or an authorized vendor and consider new units, no contingency has been applied.

Table 21-3: Mining Expansion Capital Estimate

		Unit	LOM	2021	2022	2023
Equipment	AD30 Haul Truck	$ '000s	1,256	-	628	628
	R1600G LHD	$ '000s	1,167	-	583	583
	2.5 yd LHD	$ '000s	1,081	-	540	540
	Minecat Explosives Loader	$ '000s	315	-	158	158
	Minecat with Backhoe	$ '000s	237	-	118	118
	Minecat with Loader	$ '000s	228	-	114	114
	CAT Telehandler	$ '000s	200	-	100	100
	Toyota Landcruiser	$ '000s	94	-	47	47
	Kubota	$ '000s	39	-	19	19
Grand Total		**$ '000s**	**4,615**	-	**2,307**	**2,307**

21.1.2 Process Plant Capital Cost

The process plant expansion capital required to increase throughput from 800 t/d to 1,200 t/d is presented in Table 21-4. The expansion can be summarized as follows:

- Replacing existing secondary cone crusher with a larger unit
- Additional 550 kW ball mill in series with existing primary ball mill
- Additional cyclone bank to ensure that there are always two cyclones in operation
- Two additional rougher flotation cells
- One new cleaner flotation cell
- Additional tailings thickener
- Expanded gold room.

Detailed design and major equipment ordering will commence in H2 2021 with the bulk of the construction and expenditure taking place in 2022.

Table 21-4: Process Plant Expansion Capital Estimate

		Unit	LOM	2021	2022
Plant Expansion	Primary Crushing	$ '000s	1,516	170	1,346
	Grinding	$ '000s	2,168	560	1,608
	Gold Room	$ '000s	423	-	423
	Floatation	$ '000s	1,350	120	1,230
	Reagents	$ '000s	50	-	50
	Tailings	$ '000s	800	100	700
	Process Water	$ '000s	221	-	221
	Services	$ '000s	62	-	62
	Auxiliary Equipment	$ '000s	126	-	81
	Civil & Earthworks	$ '000s	1,265	-	1,265
	Structural Steel	$ '000s	1,423	-	1,423
	Piping & Valves	$ '000s	891	-	891
	Electrical & Instr.	$ '000s	2,744	456	2,288
	EPCM/Indirects/Contin.	$ '000s	3,307	785	2,522
Grand Total		**$ '000s**	**16,347**	**2,236**	**14,111**

21.1.3 Paste Plant Capital Cost

Table 21-5 shows the sustaining capital cost for the paste plant upgrades and underground distribution system. Upgrades to the current paste backfill plant consist of the following:

- Replacing of the current undersized ceramic disk filter unit with a larger cloth disk unit
- Installing of a positive displacement pump in the process plant to pressurize the underground distribution system ensuring paste can be delivered to all areas underground.

The underground distribution system includes:

- Two cased boreholes from surface to the 100 Level RAR connection drift
- Two cased boreholes from the 100 Level RAR connection drift to the 200 Level ramp connection drift
- Main distribution underground piping, valving, and control/monitoring systems
- Surface piping from process plant to boreholes
- A positive-displacement pump booster station located underground to deliver paste to the Middle Zone.

Table 21-5: Paste Backfill Plant Upgrade and Underground Distribution System Capital Cost Estimate

		Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Pastefill Plant	Civil Works	$ '000s	420	240	180	-	-	-	-	-	-	-
	Mechanical Equipment	$ '000s	2,774	1,381	1,393	-	-	-	-	-	-	-
	P&ID	$ '000s	1,633	341	1,292	-	-	-	-	-	-	-
	EPCM + Contingency	$ '000s	1,934	857	1,077	-	-	-	-	-	-	-
	Total	**$ '000s**	**6,761**	**2,819**	**3,942**	-	-	-	-	-	-	-
Mid Zone Booster Station	Civil Works	$ '000s	458	-	273	185	-	-	-	-	-	-
	Mechanical Equipment	$ '000s	1,567	-	1,336	231	-	-	-	-	-	-
	P&ID	$ '000s	892	-	509	383	-	-	-	-	-	-
	EPCM + Contingency	$ '000s	1,347	-	999	348	-	-	-	-	-	-
	Total	**$ '000s**	**4,264**	-	**3,117**	**1,147**	-	-	-	-	-	-
Pastefill UDS Initial Capital	P&ID	$ '000s	3,342	-	3,342	-	-	-	-	-	-	-
	EPCM + Contingency	$ '000s	530	-	530	-	-	-	-	-	-	-
	Total	**$ '000s**	**3,872**	-	**3,872**	-	-	-	-	-	-	-
Pastefill UDS Sustaining Capital		$ '000s	11,224	-	-	2,090	1,691	1,684	1,790	1,620	1,433	916
Grand Total		**$ '000s**	**26,121**	**2,819**	**10,931**	**3,237**	**1,691**	**1,684**	**1,790**	**1,620**	**1,433**	**916**

21.1.4 Tailings Capital Cost

Table 21-6 presents the tailings management and water treatment sustaining capital expenditure. The North TMF will continue to be use with a combination of slurry and dry stack storage until Q2 2025. Dam raises are planned for 2021 and 2023 to bring the North TMF to its ultimate height and storage capacity, this will maintain capacity for slurry, process water and emergency water capacity for a potential environmental design flood.

Table 21-6: Tailings Management and Water Treatment Capital Cost Estimate

			LOM	2021	2022	2023	2024	2025	2026	2027	2028
North TMF	Site Preparation	$ '000s	151	77	-	74	-	-	-	-	-
	Foundation Preparation	$ '000s	656	371	79	206	-	-	-	-	-
	Dam Construction	$ '000s	7,080	3,264	236	3,580	-	-	-	-	-
	Tailings Conveyance	$ '000s	-	-	-	-	-	-	-	-	-
	Water Management	$ '000s	1,038	292	173	329	245	-	-	-	-
	Engineering	$ '000s	810	330	270	210		-	-	-	-
	Total	**$ '000s**	**9,735**	**4,334**	**758**	**4,399**	**245**	-	-	-	-
South TMF	Site Preparation	$ '000s	620	-	-	-	529	-	48	-	43
	Foundation Preparation	$ '000s	2,229	-	-	-	1,897	-	221	-	111
	Dam Construction	$ '000s	6,458	-	-	-	3,728	-	1,628	-	1,103
	Tailings Conveyance	$ '000s	1,907	-	-	-	572	1,335	-	-	-
	Water Management	$ '000s	1,647	-	-	-	422	985	60	-	180
	Engineering	$ '000s	1,320	-	-	270	210	270	210	150	210
	Total	**$ '000s**	**14,182**	-	-	**270**	**7,359**	**2,590**	**2,167**	**150**	**1,646**
Water Treatment	Treatment Plant Upgrade	$ '000s	3,048	-	1,795	1,253	-	-	-	-	-
	Treatment Plant Expansion	$ '000s	500	-	-	-	-	500	-	-	-
	Total	**$ '000s**	**3,548**	-	**1,795**	**1,253**	-	**500**	-	-	-
Grand Total		**$ '000s**	**27,465**	**4,334**	**2,552**	**5,922**	**7,603**	**3,090**	**2,167**	**150**	**1,646**

Construction of the new South TMF will begin in 2024 which will be used for slurry tailings for the remainder of the mine life.

21.1.5 Infrastructure Capital Cost

Table 21-7 presents the surface infrastructure sustaining capital cost estimate. In 2021 a new 120-person camp will be constructed in White River to accommodate the increased personnel requirement for the throughput expansion, and to provide improved hygiene conditions.

Table 21-7: Surface Infrastructure Sustaining Capital Cost Estimate

	Unit	LOM	2021	2022
WR Camp Upgrades	$ '000s	2,300	2,300	-
Heavy Equipment Shop	$ '000s	1,300	-	1,300
Surface Equipment	$ '000s	150	150	-
Other	$ '000s	91	91	-
Grand Total	**$ '000s**	**3,841**	**2,541**	**1,300**

A new surface heavy equipment workshop will be constructed in the portal area in 2022.

21.1.6 Basis of Estimate

21.1.6.1 Accuracy

The capital cost estimate has been prepared to a ±15% level of accuracy suitable for a feasibility study.

21.1.6.2 Quantity Development, and Bulk Material/Equipment Pricing

Quantities captured in the cost estimate were derived from the following:

- Quantity Take Offs (QTO)—neat quantities were derived from detailed QTOs based on project engineering designs
- Equipment lists—quantities were derived from mechanical and electrical equipment list
- Estimated quantities—known quantities not specifically identified via QTOs or lists were derived from drawings using estimating techniques
- Allowances—known quantities not specifically identified via any of the above listed methods were derived using estimating techniques
- Factors and allowances were applied to neat quantities, to determine total estimated quantities for costing purposes.

21.1.6.3 Earthworks and Civil

Earthworks and civil quantities are derived from the site layout drawings, the scope for which can generally be described as follows:

- Clearing, grubbing, and bulk earthworks for tailings dam construction and plant upgrades
- Onsite trenching, ditching, liners, geofabric/geogrid, and drainage culvert installation
- The estimate assumes that an independent service contractor will provide labour, equipment, and materials required to produce the necessary volumes of granular material needed for construction.

21.1.6.4 Structural Steel

Structural steel QTOs have been developed by engineering using project designs. They were developed using the following structural steel weight categories:

- Light Steel (0–25 kg/m)
- Medium (26–65 kg/m)
- Heavy (66–110 kg/m)
- Extra Heavy (>111 kg/m).

21.1.6.5 Mechanical and Mobile Equipment

Major mechanical/electrical/instrumentation equipment has been identified and quantified using project designs and specifications. Each piece of equipment has been identified either in the master equipment list or the QTO along with specifications and unique identifiers for reference.

All major mechanical equipment costs associated with the process plant, paste backfill plant, or underground mobile equipment fleet are supported by budgetary quotations, and they have been outlined in the mechanical/electrical/instrumentation equipment lists.

Surface contractors are expected to provide mobile equipment as part of the execution of their scopes of work.

21.1.6.6 Piping (including Civil Piping)

Piping quantities were estimated using linear metre measurements of the proposed pipe runs for each specification and size, derived from project designs and piping and instrumentation diagram (P&IDs).

21.1.6.7 Electrical and Instrumentation (Including Cabling)

Major electrical and instrumentation equipment quantities were estimated using project designs and specifications.

Electrical and instrumentation cabling quantities were estimated using linear metre measurements of the proposed cable runs for each cable specification and size, derived from project designs and SLDs.

21.1.6.8 Mining

Detailed LOM plans and schedules have been prepared by Harte Gold and Deswik to define the capital development requirements for the LOM. Harte Gold crews will perform all underground development and construction. Development costs have been developed using a zero-based methodology, validated based on historical experience at the mine.

21.1.6.9 Capital Cost Estimate Basis

Table 21-8 presents the basis of the capital cost estimate in the major project areas.

Table 21-8: Capital Cost Estimate Basis

Area	Description	Basis
Mine	Mining Method	Detailed
	Mine Plan, Layout and Infrastructure	Detailed
	Mine Production Schedule	Detailed
	Fleet Size	Detailed
	Personnel Plan	Detailed
	Backfill	Detailed
	Ventilation	Detailed
	Electrical	Detailed
	Pumping and Water Supply	Detailed
	Compressed Air	Detailed
Process	Process Design Criteria	Detailed
	Process Flow Diagrams	Detailed
	Mechanical Equipment List	Detailed
	Mass and Water Balance	Detailed
	Piping and Instrumentation Diagrams	Preliminary
	Control Philosophy	Preliminary
Plant Engineering	Engineering Design Criteria	Detailed
	Site Location	Finalized
	Geotechnical	Preliminary
	Plant General Arrangement Drawings	Detailed
	Architectural Drawings	Preliminary
	Electrical Load List	Detailed
	Electrical Design	Preliminary
	Single Line Diagrams	Preliminary
	Instrumentation Design	Preliminary
Infrastructure	Tailings North TMF	Detailed
	Tailings—South TMF	Preliminary
	Waste Dumps	Preliminary
	Roads	Preliminary

21.1.6.10 Contingency

Table 21-9 presents a summary of the contingency applied in key areas of the project. Specific contingencies were applied based on the level of engineering and design carried out in the feasibility study.

Table 21-9: Summary of Contingency Levels for Capital Projects

Project	Contingency Applied (%)
Process Plant Expansion	6
Paste Backfill Plant	11
TMF Construction	20
Water Treatment	30

21.2 Operating Cost Estimate

21.2.1 Operating Cost Summary

Operating expenses were calculated using the Sugar Zone Mine's 2021 budget as a reference point where applicable and developed from first principles when budgetary items were not available. The 2021 budget costs were adjusted to reflect increases related to mining and processing ore at higher rates as well as to incremental increases in mining unit costs as the average depth of mining increases over the LOM. The fixed and variable components of OPEX were analysed to accurately reflect the expected OPEX profile at higher tonnage throughput.

Harte Gold also received input in key areas from the following companies:

- Mining—Stantec
- Processing—Halyard
- Paste backfill—P&C
- Tailings management—Wood
- Water treatment—BQE.

Table 21-10 presents a summary of total and unit OPEX for the LOM. Costing was carried out using a zero-based methodology, validated and modified based on historical experience where applicable.

Processing and site G&A show a significant reduction in unit costs when the production rate increases from 800 t/d to 1,200 t/d in 2023. This reduction is not reflected in mining cost unit rates due to total cost increases as a result of paste backfill and increased haulage distance.

Table 21-10: Operating Cost Estimate Summary

		Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Physicals	Ore Mined	t '000s	3,454	295	317	432	431	423	431	428	416	280
	Ore Processed	t '000s	3,454	292	292	438	438	438	432	428	416	280
Costs	Mining	$ '000s	419,890	35,498	39,514	51,359	51,537	52,491	52,578	51,871	48,310	36,730
	Processing	$ '000s	121,965	12,587	11,589	14,999	15,005	14,761	14,302	14,265	14,146	10,311
	Site G&A	$ '000s	162,531	16,858	17,006	18,564	18,661	18,835	19,151	19,151	19,102	15,204
	Total	**$ '000s**	**704,385**	**64,944**	**68,109**	**84,922**	**85,204**	**86,087**	**86,031**	**85,287**	**81,557**	**62,245**
Unit Costs	Mining	$/t	122	120	124	119	120	124	122	121	116	131
	Processing	$/t	36	43	40	34	34	34	33	33	34	37
	Site G&A	$/t	48	58	58	42	43	43	44	45	46	54
	Total	**$/t**	**205**	**221**	**222**	**196**	**197**	**201**	**199**	**199**	**196**	**222**

21.2.2 Mining Cost

Table 21-11 presents the LOM personnel compliment in the mine operations and maintenance departments. Increases are projected in 2023 and 2024 as a result of the tonnage increase, additional equipment, and paste backfill implementation.

Table 21-11: Mine Operations and Maintenance Personnel

	2021	2022	2023	2024	2025	2026	2027	2028	2029
Mine Operations	***130***	***130***	***159***	***163***	***167***	***167***	***167***	***167***	***108***
Mine Superintendent	2	2	2	2	2	2	2	2	2
Mining Supervisor	9	9	9	9	9	9	9	9	4
Jumbo Driller	8	8	8	8	8	8	8	8	4
Bolter	24	24	24	24	24	24	24	24	12
Loader Face/Prep	8	8	8	8	8	8	8	8	–
Sill Miner	16	16	16	16	16	16	16	16	8
Alimak Miner	4	4	4	4	4	4	4	4	4
Truck Driver	16	16	20	24	28	28	28	28	20
Remote/Backfill	16	16	24	24	24	24	24	24	16
Scoop Operator	12	12	16	16	16	16	16	16	12
Construction	8	8	8	8	8	8	8	8	8
Grader Operator	2	2	2	2	2	2	2	2	2
Dry Person	2	2	2	2	2	2	2	2	2
Nipper	2	2	4	4	4	4	4	4	2
Mine Clerk	1	1	2	2	2	2	2	2	2
Pastefill Coordinator			2	2	2	2	2	2	2
UG Pastefill Operator			8	8	8	8	8	8	8
Mine Maintenance	***35***	***35***	***41***	***43***	***43***	***43***	***43***	***43***	***35***
Maintenance Super.	1	1	1	1	1	1	1	1	1
Lead Electrician	1	1	1	1	1	1	1	1	1
Maintenance Planner	1	1	1	1	1	1	1	1	1
Electrical Supervisor	4	4	4	4	4	4	4	4	4
Mechanical Supervisor	4	4	4	4	4	4	4	4	4
Journeyman Electrician	4	4	6	8	8	8	8	8	4
Apprentice Electrician	4	4	4	4	4	4	4	4	4
Heavy Duty Mechanic	4	4	6	6	6	6	6	6	4
Journeyman Mechanic	6	6	8	8	8	8	8	8	6
Apprentice Mechanic	2	2	2	2	2	2	2	2	2
Apprentice Mechanic 4	4	4	4	4	4	4	4	4	4

Personnel costs have been estimated using base salary, overtime, bonus and burden to provide a total cost to company. Table 21-12 summarizes the total personnel costs for mining and maintenance, and contains the total personnel cost, of which approximately 35% are allocated to sustaining capital development costs and are not reflected in the OPEX.

Table 21-12: Mine Personnel Cost

Mine Personnel Cost		LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Mining Operations (direct and indirect)	$ '000s	198,796	19,355	19,355	23,322	23,826	24,330	24,330	24,330	24,330	15,618
Maintenance (indirect)	$ '000s	63,584	6,196	6,196	7,216	7,556	7,556	7,556	7,556	7,556	6,196
Total	**$ '000s**	**262,380**	**25,551**	**25,551**	**30,538**	**31,382**	**31,886**	**31,886**	**31,886**	**31,886**	**21,814**

The direct mining cost categories are:

- Ore development
- Longhole drill and blast
- Stoping
- Backfill
- Haulage.

Direct mining costs are estimated on a unit-cost basis for each mining activity using the appropriate combination of the following items for each activity:

- Operating personnel
- Consumables
  - Bits and steel
  - Explosives
  - Ground-support supplies
  - Ventilation supplies
  - Piping and valving.
- Equipment
  - Fuel and lube
  - Maintenance consumables.

Indirect costs are based on a monthly total cost estimate for mining, maintenance, and technical personnel, power, mine heating, and geological infill drilling.

Table 21-13 presents a summary of the mine OPEX over the LOM.

Table 21-13: Mine Operating Cost

		Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Physicals	Ore Mined	t '000s	3,454	295	317	432	431	423	431	428	416	280
	Waste Mined	t '000s	2,088	286	249	234	252	246	249	240	229	102
	Operating Development	m	36,126	3,752	4,668	4,718	4,761	4,512	4,154	3,829	4,242	1,490
Direct Cost	Ore Development	$ '000s	65,581	5,048	8,657	8,753	9,199	8,114	7,868	7,000	7,973	2,969
	LH Drill & Blast	$ '000s	114,360	9,277	9,517	14,083	13,300	14,962	14,884	14,977	12,530	10,829
	Stoping Costs	$ '000s	16,657	2,300	1,238	1,784	2,048	1,869	1,816	1,857	2,431	1,313
	Backfill	$ '000s	21,739	937	557	3,022	2,977	3,115	2,978	2,850	2,668	2,636
	Haulage	$ '000s	35,722	1,744	2,605	3,894	4,208	4,517	5,037	5,001	4,705	4,010
	Total Direct Mining Cost	**$ '000s**	**254,059**	**19,307**	**22,574**	**31,537**	**31,731**	**32,578**	**32,584**	**31,685**	**30,307**	**21,756**
Indirect Costs	Mining Indirect Labour	$ '000s	29,750	3,007	2,858	3,558	3,459	3,471	3,477	3,516	3,539	2,866
	Eng. Labour	$ '000s	7,137	622	654	847	824	826	828	837	842	857
	Eng. Materials	$ '000s	923	115	115	115	115	115	115	115	87	29
	Geo Labour	$ '000s	12,418	1,154	1,473	1,473	1,473	1,473	1,473	1,473	1,213	1,213
	Geo Infill Drilling	$ '000s	21,306	2,841	2,841	2,841	2,841	2,841	2,841	2,841	1,420	
	Geo Other	$ '000s	3,392	424	424	424	424	424	424	424	318	106
	Mine Maintenance Labour	$ '000s	39,809	3,062	3,471	4,680	4,765	4,780	4,788	4,843	4,874	4,546
	Mine Maintenance Other	$ '000s	8,925	1,364	770	1,176	1,155	1,170	1,185	1,212	691	201
	Power	$ '000s	13,624	800	1,227	1,443	1,510	1,570	1,619	1,671	1,760	2,024
	Propane Heating	$ '000s	7,083	419	722	880	855	858	859	869	875	746
	UG General Fixed Costs	$ '000s	13,377	1,486	1,486	1,486	1,486	1,486	1,486	1,486	1,486	1,486
	Other Fixed Costs	$ '000s	8,086	898	898	898	898	898	898	898	898	898
	Total Indirect Mining Cost	**$ '000s**	**165,831**	**16,192**	**16,940**	**19,823**	**19,806**	**19,914**	**19,995**	**20,186**	**18,003**	**14,973**
Total Mining Cost		**$ '000s**	**419,890**	**35,498**	**39,514**	**51,359**	**51,537**	**52,491**	**52,578**	**51,871**	**48,310**	**36,730**
Unit Mining Cost		**$/t**	**122**	**120**	**124**	**119**	**120**	**124**	**122**	**121**	**116**	**131**

21.2.3 Processing Cost

Table 21-14 presents the LOM personnel compliment in the processing department. Increased plant throughput and operation of the paste backfill plant in 2023 results in an increased personnel requirement.

Table 21-14: Processing Operating Personnel

	2021	2022	2023	2024	2025	2026	2027	2028	2029
Mill	***46***	***46***	***54***	***54***	***54***	***54***	***54***	***54***	***46***
Plant Manager	1	1	1	1	1	1	1	1	1
Mill General Foreman	1	1	1	1	1	1	1	1	1
Metallurgist	1	1	1	1	1	1	1	1	1
Shift Supervisors	4	4	4	4	4	4	4	4	4
Crusher Operators	4	4	4	4	4	4	4	4	4
Plant Operators	4	4	8	8	8	8	8	8	4
Dry stack/Paste Op.	4	4	4	4	4	4	4	4	4
ETP Operators	4	4	4	4	4	4	4	4	4
Control Room Op.	4	4	4	4	4	4	4	4	4
Mill Labourer	4	4	4	4	4	4	4	4	4
Mechanical/Maintenance	4	4	6	6	6	6	6	6	4
Planner/Supervisor	2	2	2	2	2	2	2	2	2
Electrical/Maintenance	4	4	6	6	6	6	6	6	4
Chief Assayer	1	1	1	1	1	1	1	1	1
Technician	4	4	4	4	4	4	4	4	4

Personnel costs have been estimated using base salary, overtime, bonus, and burden to provide a total cost to company. Table 21-12 summarizes the total personnel costs for mining and maintenance.

Table 21-15: Processing Personnel Operating Cost

	Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Personnel Cost	$ ‘000s	67,373	6,766	6,763	7,884	7,884	7,884	7,884	7,884	7,884	6,538

Fixed process-plant costs are based on a monthly total cost estimate for personnel, maintenance spares, propane, and various other miscellaneous items.

The variable processing cost categories are:

- Power
- Reagents
- Mill balls
- Mill liners
- Crusher liners
- Dry stack placement.

Variable costs are zero-based estimated on a unit cost basis for each item and applied to the LOM processing plan.

Table 21-16 presents the processing OPEX for the LOM.

Table 21-16: Processing Operating Cost

		Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Physicals	Ore Processed	t '000s	3,454	292	292	438	438	438	432	428	416	280
Fixed Costs	Plant Ops. Personnel	$ '000s	62,803	6,267	6,263	7,335	7,335	7,335	7,335	7,335	7,335	6,263
	Contract Maintenance	$ '000s	4,570	500	500	549	549	549	549	549	549	275
	Analytical Supplies	$ '000s	1,826	102	209	226	226	226	226	226	226	158
	Safety Equipment	$ '000s	1,692	114	114	225	225	225	225	225	225	113
	Plant Maintenance Spares	$ '000s	8,311	1,129	912	1,003	1,003	1,003	1,003	1,003	1,003	251
	E&I Maintenance Spares	$ '000s	821	86	86	104	104	104	104	104	104	26
	Piping	$ '000s	216	23	23	26	26	26	26	26	26	13
	Lubricants Allowance	$ '000s	736	80	80	88	88	88	88	88	88	44
	Gasoline & Propane	$ '000s	1,733	198	198	198	198	198	198	198	198	149
	ETP & Fire Water Reagents	$ '000s	1,801	111	111	220	220	228	228	228	228	228
	Freight/Courier Charges	$ '000s	1,005	103	103	123	123	123	123	123	123	62
	Paste Plant	$ '000s	2,498	413	413	253	253	253	253	253	253	152
	Others	$ '000s	1,086	115	115	132	132	132	132	132	132	66
	Total Fixed Costs	**$ '000s**	**89,096**	**9,240**	**9,126**	**10,484**	**10,484**	**10,491**	**10,491**	**10,491**	**10,491**	**7,798**
Variable Costs	Power	$ '000s	15,402	1,870	984	1,985	1,985	1,949	1,835	1,813	1,752	1,230
	Reagents	$ '000s	4,140	359	359	522	522	522	515	511	496	334
	Mill Balls	$ '000s	7,724	435	435	1,046	1,046	1,046	1,031	1,023	993	669
	Mill Liners	$ '000s	2,022	110	110	275	275	275	271	269	261	176
	Crusher Liners	$ '000s	1,274	109	109	161	161	161	159	158	153	103
	Dry Stack Placement	$ '000s	2,308	466	467	526	533	316				
	Total Variable Costs	**$ '000s**	**32,869**	**3,348**	**2,464**	**4,515**	**4,521**	**4,270**	**3,811**	**3,773**	**3,654**	**2,512**
Total Processing Cost		**$ '000s**	**121,965**	**12,587**	**11,589**	**14,999**	**15,005**	**14,761**	**14,302**	**14,265**	**14,146**	**10,311**
Unit Processing Cost		**$/t**	**35**	**43**	**40**	**34**	**34**	**34**	**33**	**33**	**34**	**37**

21.2.4 G&A Cost

Table 21-16 presents the G&A personnel schedule for the LOM. The increase in throughput from 800 t/d to 1,200 t/d does not require a significant increase in G&A personnel.

Table 21-17: G&A Personnel

	2021	2022	2023	2024	2025	2026	2027	2028	2029
Geology	***9***	***10***	***10***	***10***	***10***	***10***	***10***	***8***	***8***
Chief Mine Geologist	1	1	1	1	1	1	1	1	1
Senior Geologist	1	2	2	2	2	2	2	2	2
Underground GC Geo.	4	4	4	4	4	4	4	4	4
Core Logging Geo.	2	2	2	2	2	2	2	–	–
Geologic Technician	1	1	1	1	1	1	1	1	1
Engineering	***7***	***7***	***8***	***8***	***8***	***8***	***8***	***8***	***7***
Chief Engineer	1	1	1	1	1	1	1	1	1
Senior Mine Engineer	2	2	2	2	2	2	2	2	2
Mine Engineer	2	2	2	2	2	2	2	2	2
Mine Technician	–	–	1	1	1	1	1	1	–
UG Surveyor	2	2	2	2	2	2	2	2	2
Site Surface	***16***	***16***	***18***	***18***	***18***	***18***	***18***	***18***	***16***
Surface Super.	1	1	1	1	1	1	1	1	1
Surface Lead Hand	1	1	1	1	1	1	1	1	1
Heavy Equipment Op.	8	8	8	8	8	8	8	8	8
Labourer	4	4	4	4	4	4	4	4	4
Additional Labourers	2	2	4	4	4	4	4	4	2
Safety	***4***	***4***	***5***	***5***	***5***	***5***	***5***	***5***	***4***
H&S Coordinator	2	2	2	2	2	2	2	2	2
OHN	1	1	1	1	1	1	1	1	1
UG Safety/Trainer	1	1	2	2	2	2	2	2	1
Environmental	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***
Environmental Coord.	1	1	1	1	1	1	1	1	1
Environmental Tech.	1	1	1	1	1	1	1	1	1
Site Admin	***15***	***15***	***15***	***15***	***15***	***15***	***15***	***15***	***13***
Admin Manager	1	1	1	1	1	1	1	1	1
Site Controller	1	1	1	1	1	1	1	1	1
Accounting Payable	3	3	3	3	3	3	3	3	2
Payroll /Admin	3	3	3	3	3	3	3	3	2
Site Controller Assist.	1	1	1	1	1	1	1	1	1
Procurement Coord.	4	4	4	4	4	4	4	4	4
Materials Handler	2	2	2	2	2	2	2	2	2
Human Resources	***5***	***5***	***5***	***5***	***5***	***5***	***5***	***5***	***5***
HR Superintendent	1	1	1	1	1	1	1	1	1
HR Specialist	1	1	1	1	1	1	1	1	1
Housekeeping	1	1	1	1	1	1	1	1	1
Janitor	2	2	2	2	2	2	2	2	2

Table 21-18 presents the G&A costs for the LOM.

Table 21-18: G&A Operating Cost

		Unit	LOM	2021	2022	2023	2024	2025	2026	2027	2028	2029
Physicals	Ore Processed	t '000s	3,454	292	292	438	438	438	432	428	416	280
Site Surface Labour		$ 000	17,105	1,733	1,779	1,969	1,969	1,969	1,969	1,969	1,969	1,779
Site Surface		$ 000	29,656	2,771	2,775	3,496	3,490	3,614	3,930	3,930	3,930	1,717
Accommodations Costs		$ 000	44,439	4,615	4,615	5,082	5,181	5,230	5,230	5,230	5,181	4,074
Safety and Security Labour		$ 000	7,475	955	680	860	860	860	860	860	860	680
Safety and Security—Other 2012		$ 000	13,455	1,495	1,495	1,495	1,495	1,495	1,495	1,495	1,495	1,495
Environmental Labour		$ 000	2,387	259	266	266	266	266	266	266	266	266
Environmental and permits		$ 000	4,518	499	499	499	504	504	504	504	504	504
Site Admin Labour		$ 000	18,890	1,809	2,161	2,161	2,161	2,161	2,161	2,161	2,161	1,954
Site admin/IT—Other		$ 000	14,966	1,663	1,663	1,663	1,663	1,663	1,663	1,663	1,663	1,663
Human Resources Labour		$ 000	5,558	606	619	619	619	619	619	619	619	619
HR–Other		$ 000	4,084	454	454	454	454	454	454	454	454	454
Total Site G&A Cost		**$ 000**	**162,531**	**16,858**	**17,006**	**18,564**	**18,661**	**18,835**	**19,151**	**19,151**	**19,102**	**15,204**
Unit G&A Cost		**$/t**	**47**	**58**	**58**	**42**	**43**	**43**	**44**	**45**	**46**	**54**

22 ECONOMIC ANALYSIS

An economic model was developed to estimate annual cash flows and sensitivities for the Sugar Zone Mine. Cash flows were calculated on a post-tax basis at the asset level, and exclude corporate level costs and capital structure adjustments.

Sensitivity analyses were performed for variation in gold price, foreign-exchange rate, mine costs, expansion capital, and sustaining capital to determine their relative importance as value drivers.

The estimates of CAPEX and OPEX were built up from a fundamental cost base approach and are summarized in Section 16.

22.1 Assumptions

All costs and economic results are reported in Canadian dollar nominal terms, unless otherwise noted. Table 22-1 outlines the planned LOM tonnage and grade estimates.

A total of 849,000 oz is expected to be recovered over the LOM at an average diluted head grade of 7.1 g/t Au. LOM gold production and head grade are summarized in Table 22-1.

Table 22-1: LOM Plan Summary

	Unit	2021 800 t/d	2023 1,200 t/d	LOM Average Years 2023 to 2027	LOM Total Years 2021 to 2029
Tonnes Processed	t	292,000	438,000	434,831	3,454,074
Grade	g/t Au	7.1	7.7	7.5	7.2
Recovery	%	94.0	94.4	94.4	94.3
Gold Recovered	oz Au	62,458	102,261	98,743	751,601

This Feasibility Study was based on analyst consensus forward price estimates incorporating 31 broker estimates and a fixed US$:C$ conversion, outlined in Table 22-2.

Table 22-2: Macroeconomic Assumptions

	Unit	2021	2022	2023	2024	2025
Gold Price	US$/oz	1,938	1,871	1,782	1,735	1,624
Exchange Rate	C$:US$	1.31:0.76	1.31:0.76	1.31:0.76	1.31:0.76	1.31:0.76

Other economic factors and assumptions used in the economic analysis include the following:

- All figures are before hedge payments and corporate level debt.
- Cash flows are calculated for the period in which they are incurred, and are not adjusted for incoming or outgoing payments.
- Discount rate of 5%.
- Closure costs of $8.2 million paid over 2030 to 2039.

22.2 Net Revenue

Mine revenue is derived from the sale of gold doré and concentrate into the international marketplace. The mine has contractual arrangements for refining gold doré, and smelting and refining gold concentrate. These contracts have been factored in all revenue forecasts.

Figure 22-1 illustrates the annual recovered gold and cumulative recovered gold by year.

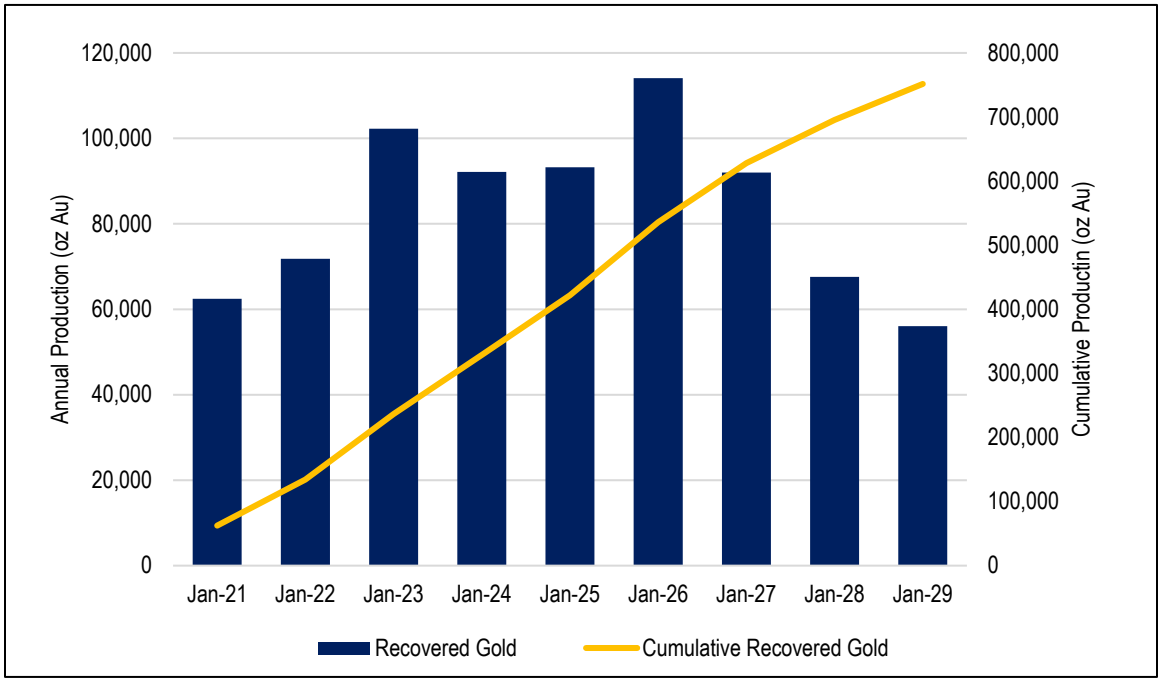


Figure 22-1: Annual and Cumulative Gold Production

22.3 Taxes

The Sugar Zone Mine is subject to government taxes based on the following:

- Ontario Mining Tax—10%
- Federal and provincial income tax—25%
- The economic analysis factors in existing tax pools.

Based on the assumed tax rate and pre-tax financial projections in the model, the Sugar Zone Mine is expected incur mining taxes of $109 million over a nine-year mine life, implying an average effective tax rate of 21%.

22.4 Economic Analysis

The Sugar Zone Mine is economically viable, with a net present value at a 5% discount rate ($NPV_{5\%}$) of $417 million (pre-tax) and $332 million (post-tax). An internal rate of return (IRR) of 89% was calculated on the differential post-tax cash flow between operating at steady state of 800 t/d and expanding to 1,200 t/d in 2023.

Table 22-3 and Figure 22-2 show the detailed economic results of this evaluation.

Table 22-3 Summary of Economic Results

	Unit	2021 800 t/d	2023 1,200 t/d	LOM Average Years 2023 to 2027	LOM Total Year 2021 to 2029
Cash Flow					
Net Revenues[1]	$M	149	225	204	1,582
Mine EBITDA[2]	$M	84	140	119	878
Pre-Tax Mine Free Cash Flow[3]	$M	36	96	80	512
Post-Tax Mine Free Cash Flow[4]	$M	35	88	61	404
NPV and IRR					
Pre-Tax $NPV_{5\%}$	$M	-	-	-	417
Post-Tax $NPV_{5\%}$	$M	-	-	-	332
IRR vs. Current 800 t/d[5]	%	-	-	-	89

Notes: [1] Revenues after transportation and refining charges, excludes corporate costs and hedge payments.
[2] Net revenues less production costs, excludes hedge payments, interest, tax, capital, and corporate costs.
[3] Operating cash flow net of capital, tax, and NPI, excludes corporate costs and hedge payments.
[4] Operating cash flow after Ontario Mining Tax and income tax.
[5] IRR is calculated on a differential post-tax cash flow basis between 1,200 t/d expansion and operating at 800 t/d.

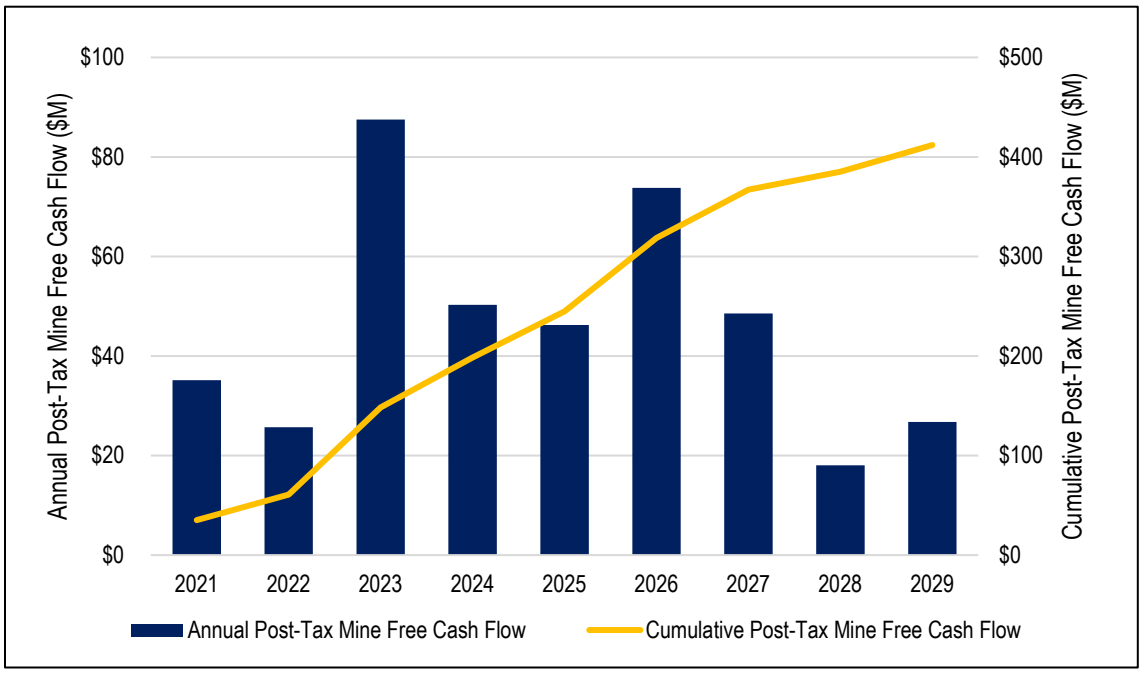


Figure 22-2: Cumulative Projected Cash Flows at a Range of Price Scenarios

22.5 Sensitivities

A sensitivity analysis was performed to test value drivers on the NPV, holding the 5% discount rate constant. The results of this analysis are provided in Figure 22-3. The $NPV_{5\%}$ is most sensitive to gold price, foreign exchange, and mine costs.

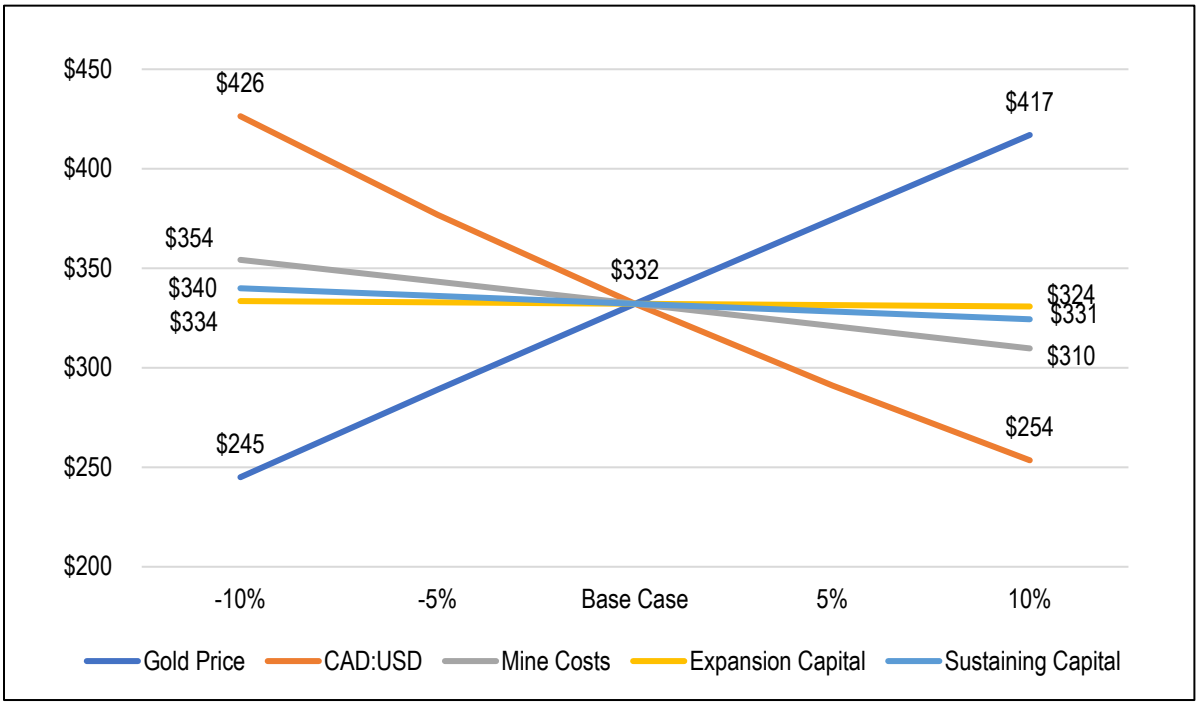


Figure 22-3: Post-Tax $NPV_{5\%}$ Sensitivity

22.6 Financial Model Summary

The financial model summary is shown in Table 22-8.

Table 22-4: Financial Model Summary

	Unit	LOM Dec-20	2021 Dec-21	2022 Dec-22	2023 Dec-23	2024 Dec-24	2025 Dec-25	2026 Dec-26	2027 Dec-27	2028 Dec-28	2029 Dec-29	2030 Dec-30	2031 Dec-31	2032 Dec-32	2033 Dec-33	2034 Dec-34	2035 Dec-35	2036 Dec-36	2037 Dec-37	2038 Dec-38	2039 Dec-39
Macro																					
Gold Price	t	***1,709***	1,938	1,871	1,782	1,735	1,624	1,624	1,624	1,624	1,624	1,624	1,624	1,624	1,624	1,624	1,624	1,624	1,624	1,624	1,624
Exchange Rate	C$:US$	***0.7600***	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600	0.7600
Mine Production																					
Mined Tonnes	t	***3,454,074***	295,034	317,426	432,003	430,741	423,412	431,171	428,369	415,682	280,235	-	-	-	-	-	-	-	-	-	-
Ore Tonnes	t	***3,454,074***	292,000	292,000	438,000	438,000	438,000	431,788	428,369	415,682	280,235	-	-	-	-	-	-	-	-	-	-
Throughput	t/d	***1,051***	*800*	*800*	*1,200*	*1,197*	*1,200*	*1,183*	*1,174*	*1,136*	*768*	-	-	-	-	-	-	-	-	-	-
Processed Grade	g/t	***7.2***	7.1	8.1	7.7	6.9	7.0	8.7	7.1	5.4	6.6	-	-	-	-	-	-	-	-	-	-
Recovery	%	***94.3***	94.0	94.0	94.4	94.4	94.4	94.4	94.4	94.4	94.4	-	-	-	-	-	-	-	-	-	-
Recovered Gold	oz	***751,601***	62,458	71,812	102,261	92,136	93,239	114,085	91,997	67,570	56,044	-	-	-	-	-	-	-	-	-	-
Cash Flow																					
Net Revenues	**$ '000s**	***1,581,889***	**148,759**	**165,338**	**225,198**	**197,187**	**186,642**	**228,956**	**184,122**	**134,540**	**111,146**	-	-	-	-	-	-	-	-	-	-
Operating Costs																					
Mine Production	$ '000s	***(419,890)***	(35,498)	(39,514)	(51,359)	(51,537)	(52,491)	(52,578)	(51,871)	(48,310)	(36,730)	-	-	-	-	-	-	-	-	-	-
Mill Costs	$ '000s	***(121,965)***	(12,587)	(11,589)	(14,999)	(15,005)	(14,761)	(14,302)	(14,265)	(14,146)	(10,311)	-	-	-	-	-	-	-	-	-	-
Site G&A	$ '000s	***(162,531)***	(16,858)	(17,006)	(18,564)	(18,661)	(18,835)	(19,151)	(19,151)	(19,102)	(15,204)	-	-	-	-	-	-	-	-	-	-
Mine EBITDA	**$ '000s**	***877,504***	***83,816***	***97,229***	***140,276***	***111,984***	***100,556***	***142,924***	***98,835***	***52,983***	***48,901***	-	-	-	-	-	-	-	-	-	-
Capital Development																					
Mine Development	$ '000s	***(199,229)***	(25,602)	(22,081)	(21,333)	(24,105)	(22,785)	(23,675)	(24,346)	(22,275)	(13,026)	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	$ '000s	***(125,507)***	(18,127)	(28,104)	(16,517)	(15,209)	(12,409)	(10,741)	(6,892)	(6,617)	(2,714)	(5,490)	(946)	(706)	(464)	(95)	(96)	(95)	(95)	(95)	(95)
Expansion Capital	$ '000s	***(21,097)***	(2,517)	(16,273)	(2,307)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
First Nations																					
First Nations NPI	$ '000s	***(20,007)***	(1,176)	(1,631)	(3,847)	(2,656)	(2,364)	(4,090)	(2,453)	(713)	(1,076)	-	-	-	-	-	-	-	-	-	-
Pre-Tax Mine Free Cash Flow	**$ '000s**	***511,664***	**36,393**	**29,141**	**96,272**	**70,013**	**62,998**	**104,418**	**65,143**	**23,378**	**32,085**	**(5,490)**	**(946)**	**(706)**	**(464)**	**(95)**	**(96)**	**(95)**	**(95)**	**(95)**	**(95)**
Taxes																					
Income Tax	$ '000s	***(80,751)***	(569)	(2,581)	(6,324)	(14,997)	(13,074)	(22,599)	(12,006)	(4,295)	(4,306)	-	-	-	-	-	-	-	-	-	-
Ontario Mining Tax	$ '000s	***(27,111)***	(682)	(882)	(2,419)	(4,734)	(3,694)	(8,025)	(4,599)	(1,038)	(1,038)	-	-	-	-	-	-	-	-	-	-
Post-Tax Mine Free Cash Flow	**$ '000s**	***403,802***	**35,143**	**25,677**	**87,529**	**50,282**	**46,230**	**73,795**	**48,537**	**18,045**	**26,741**	**(5,490)**	**(946)**	**(706)**	**(464)**	**(95)**	**(96)**	**(95)**	**(95)**	**(95)**	**(95)**
Unit Costs	**%**	***21***																			
Mine Production	$/t	***122***	*120*	*124*	*119*	*120*	*124*	*122*	*121*	*116*	*131*	-	-	-	-	-	-	-	-	-	-
Processing	$/	***35***	*43*	*40*	*34*	*34*	*34*	*33*	*33*	*34*	*37*	-	-	-	-	-	-	-	-	-	-
Site G&A	$/t	***47***	*58*	*58*	*42*	*43*	*43*	*44*	*45*	*46*	*54*	-	-	-	-	-	-	-	-	-	-
Cash Operating Cost—Before Royalties	US$/oz	***737***	*823*	*750*	*651*	*725*	*724*	*591*	*727*	*948*	*881*	-	-	-	-	-	-	-	-	-	-
Cash Operating Cost—Net Royalties	US$/oz	***804***	*898*	*822*	*721*	*793*	*787*	*655*	*790*	*1,011*	*944*	-	-	-	-	-	-	-	-	-	-
AISC	US$/oz	***1,132***	*1,430*	*1,354*	*1,002*	*1,117*	*1,074*	*884*	*1,048*	*1,336*	*1,157*	-	-	-	-	-	-	-	-	-	-
NPV Analysis																					
Discount Date		-	Dec-21	Dec-22	Dec-23	Dec-24	Dec-25	Dec-26	Dec-27	Dec-28	Dec-29	Dec-30	Dec-31	Dec-32	Dec-33	Dec-34	Dec-35	Dec-36	Dec-37	Dec-38	Dec-39
Discount Period		-	0.5x	1.5x	2.5x	3.5x	4.5x	5.5x	6.5x	7.5x	8.5x	9.5x	10.5x	11.5x	12.5x	13.5x	14.5x	15.5x	16.5x	17.5x	18.5x
Discount Factor	%	-	98	93	89	84	80	76	73	69	66	63	60	57	54	52	49	47	45	43	41
Pre-Tax NPV		**417,170**	35,516	27,084	85,217	59,022	50,579	79,843	47,439	16,214	21,193	(3,453)	(567)	(403)	(252)	(49)	(47)	(45)	(43)	(41)	(39)
Post-Tax NPV		**332,158**	34,296	23,865	77,478	42,389	37,117	56,427	35,346	12,515	17,663	(3,453)	(567)	(403)	(252)	(49)	(47)	(45)	(43)	(41)	(39)

23 ADJACENT PROPERTIES

There are no material adjacent properties that are significant to the Sugar Zone Property results and interpretations.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information that is material to this report.

25 INTERPRETATION AND CONCLUSIONS

Harte Gold personnel reviewed and audited the historical exploration data available for the Sugar Zone Mine, as well as the exploration methodologies adopted to generate the data. Exploration work is professionally managed, and procedures are adopted that meet accepted industry best practices. The QP is of the opinion that the exploration data are sufficiently reliable to interpret with confidence the boundaries of the gold mineralization and support evaluation and classification of Mineral Resources in accordance with generally accepted *CIM Estimation of Mineral Resource and Mineral Reserve Best Practices Guidelines* and *CIM Definition Standards for Mineral Resources and Mineral Reserves*.

The Project's drilling database includes information from 972 drill holes comprising 355,593 m, of which 750 drill holes totalling 339,234 m were used to generate the Mineral Resource estimate.

The primary mining method used at the Sugar Zone Mine is longitudinal-retreat longhole open stoping and is deemed suitable considering the geometry of the orebody.

Harte Gold performs regular reconciliations between production and the reserve block model, and results have generally been within industry acceptable ranges, with current results skewing slightly positive. Harte Gold uses the reconciliation process to validate its Mineral Resource estimation parameters and procedures.

Harte Gold has undertaken a detailed engineering and economic study of an expansion of mine and mill throughput from 800 t/d to 1,200 t/d to mine the Mineral Reserves. Harte Gold has concluded that mining the current Mineral Reserves at an increased rate of 1,200 t/d is both feasible and economically beneficial.

The current 800 t/d mill consistently achieves recoveries of approximately 94%. The mill's expansion to 1,200 t/d includes adding a new secondary cone crusher, adding a ball mill, modifying the gravity and flotation circuits, adding a tailings thickener as well as other upgrades, additions, and expansions within the process plant.

As part of the Project, the mining mobile equipment fleet will also be expanded by adding one 30-tonne haul truck, three additional 1.8 m^3 LHDs, and additional ancillary equipment such as personnel carriers.

The existing and planned site infrastructure aside from the process plant expansion is not expected to require any material modifications to enable the expansion, although some facilities such as the South TMF or ventilation raises may need to be constructed sooner than would be the case if the expansion did not move ahead.

Feasibility-level design was completed for the North TMF and South TMF to safely store approximately 2.5 Mt of tailings from November 2020 to December 2029 (end of LOM) and accommodate site water-

management requirements. The dam designs were developed in accordance with the requirements of the LRIA (MNR, 2011) and the CDA (2013, 2019) dam safety guidelines and considers elements of the Global Tailings Review.org guidelines.

- Dams are designed as embankment dams, with low-permeability BGM liner on the upstream slope that will be tied to concrete plinth founded on treated bedrock. Bedrock grouting will be required at the North TMF east dam and South TMF north dam. the upstream and downstream design slopes of the dams are 2.5H:1V, except for the downstream slopes of the North TMF East and West Dams, where a slope of 2.2H:1V is adopted.
- Dams will be founded on approved competent native soils or bedrock and will be constructed using mainly the waste rock from underground mine operations. Sand and gravel fill and/or quarry rock will be used to supplement shortages of waste rock. The bedding sand and sand and gravel will be supplied from local borrow pits. The splitter berm will be constructed with end-dumped waste rock.
- Bedrock grouting requirements for North TMF and South TMF dams are based on available information and data. Additional site investigation should be performed to confirm and/or update the grouting requirement.

The Sugar Zone Mine is operating within environmental compliance although a number of operational permits that will need to be amended to allow for expansion. Permitting is expected to take between 12 and 18 months.

The Sugar Zone Mine has been, and will continue to be, a major contributor to the local economy. Harte Gold will continue to engage and work with area Indigenous communities and other communities of interest.

CAPEX and OPEX have been undertaken to a feasibility-study level of detail based on the scope of expansion defined in this Technical Report, and appropriate levels of rigour and contingency have been considered for the purposes of this Feasibility Study. All material and equipment quotes are current as of the effective date of this Technical Report, and there may be positive or negative variations in the actual costs at time of purchase due to market conditions.

The Project is economically viable, with an after-tax IRR of 89%, which is calculated on the differential after-tax cash flow between the scenario of expanding to 1,200 t/d and the base-case scenario of continuing to mine at 800 t/d. Post-tax $NPV_{5\%}$ is $332 million.

26 RECOMMENDATIONS

At the conclusion of the Feasibility Study for Expansion to 1,200 t/d the following recommendations are being made.

26.1.1 General

The Feasibility Study shows positive economic benefits for an expansion of the Sugar Zone Mine to 1,200 t/d throughput and the Project should move forward.

26.1.2 Geology

Harte Gold should continue definition drilling programs to upgrade the quality of the known Mineral Reserves and identify extensions within the mining area.

The LOM OPEX model considers an annual expenditure of $2.8 million for this purpose from 2021 through 2027. In the model, the definition drilling program is reduced to $1.4 million in 2028 as known Mineral Reserves become exhausted. In the probable case that the drilling programs identify additional Mineral Reserves that extend mine life, this expenditure is expected to continue at 2021 levels.

Harte Gold should continue their current focus on reconciling mined material versus the Mineral Resource block model to improve understanding of the Mineral Resource.

26.1.3 Mining

The Company should continue to improve planning and operational processes with a focus on minimizing unplanned dilution and improving productivities. These focus areas will improve the operation's OPEX profile and improve the average grade of ore mined, and will therefore have the potential to positively impact the operating margins estimated in this Feasibility Study.

Harte Gold should continue improving the detailed design of the Middle Zone; the Company will reach first ore in the Middle Zone during Q2 2021, which should help inform final optimizations around level spacing and equipment sizing to ensure safe and cost-effective mining of the new zone.

26.1.4 Processing

Harte Gold should proceed with detailed design and engineering of the process plant expansion. The only additional testwork recommended is for the sizing of the cleaner flotation cell. The Project detailed design work should be done by an engineering company that specializes in this type of engineering.

The Project CAPEX estimate includes $1.2 million for this purpose and it is planned to be spent between 2021 and 2022.

Additional testwork for the sizing of the cleaner flotation cell is expected to cost around $10,000 and is considered in the process plant's 2021 operating budget.

26.1.5 Paste Backfill

Paste backfill represents a significant portion of the LOM CAPEX and increases the mining unit cost per tonne. Given current Mineral Reserves, the LOM production plan considers implementing paste backfill beginning in 2023. Harte Gold has identified that there could be an opportunity to defer or eliminate implementing paste backfill, and that this may show a positive economic benefit. A detailed trade-off study should be undertaken to understand the impact that such a decision could have on:

- Geotechnical stability
- Mineral Reserve recovery and Mineral Reserve quantities
- Tailings storage impoundments
- Mining equipment fleet and ventilation system
- Operating and capital cost estimates.

Large portions of this study can be done internally by Harte Gold technical staff. Portions of the study, such as geotechnical modelling and analysis, and tailings impoundment design, will require the input of qualified engineers. The costs involved in this trade-off study have not been considered in this Technical Report; the Company expects that it may cost up to $200,000.

Pending results of a trade-off study, Harte Gold should proceed with detailed design and engineering of the paste backfill plant expansion. This work should be done by an engineering company that specializes in this type of engineering.

The sustaining CAPEX estimate includes $840,000 for this purpose and it is planned to be spent between 2021 and 2022.

26.1.6 Tailings Storage

Additional site investigations must be performed at both the North TMF and South TMF sites to optimize the proposed designs and to support detailed design.

- The sustaining CAPEX estimate includes $100,000 for one site investigation program at the North TMF
- The sustaining CAPEX estimate includes $200,000 for two site investigation programs at the South TMF.

- The first program will support overall optimization and starter dam detailed design. This program will also include selected test pitting to assess the feasibility and potential effectiveness of a clean water diversion around the South TMF to reduce the required water storage and discharge volumes. This program will further assess some in situ material deposits near the left dam abutment to understand whether they are suitable for use as bedding sand and/or sand and gravel fill to reduce the quantity of material required to be sourced and hauled to the construction site.
- The second program will support the Stage 2 dam raise design

The proposed South TMF design should be optimized and updated by identifying areas with shallow overburden at the left abutment, and assessing the crevasse at the valley bottom. Fill volumes and BGM liner requirements related to the camber thickness needed to compensate for potential post-construction crest settlement are not included in the quantity estimates. These optimizations are to be performed as part of detailed design, with input from the above-noted site investigations.

The sustaining CAPEX estimate includes $375,000 for detailed design and optimization of all stages of the South TMF

26.1.7 Permitting and Environmental

Harte Gold should begin the process of applying for permit amendments as described in Section 20. The permitting process is expected to take between 12 and 18 months and may involve community and Indigenous consultation as prescribed by the MENDM:

- The Project CAPEX estimate includes $136,000 for this purpose.
- Additional unbudgeted costs of up to $35,000 may be incurred in 2022 if Harte Gold is required to re-print hard copies of some closure plan certificates.

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28 CERTIFICATES OF AUTHOR

28.1 Chris McCann, P.Eng.

I, Chris McCann, P.Eng., as an Author of this report titled *National Instrument (NI) 43-101 Technical Report—Expansion to 1,200 t/d for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario, Canada* (the Technical Report), with an Effective Date of 30 September 2020 for the Mineral Resource estimate, and 31 December 2020 for the updated Mineral Reserves estimate, and dated 5 March 2021, do hereby certify that:

- I am the Director, Technical Services for Harte Gold Corp. with an office at Suite 2400 – 161 Bay St, Toronto, Ontario, M5J 2S1.
- I graduated from the University of British Columbia, Vancouver, B.C. in 2009 with a Bachelor's Degree in Mining Engineering and from IE Business School in 2019 with a Master's in Business Administration.
- I am a member in good standing of the Professional Engineers of Ontario, License #100500259. I have 12 years of relevant experience working as a Mining Engineer.
- I am a "Qualified Person" as that term is defined in NI 43-101 – Standards of Disclosure for Mineral Projects.
- I am responsible for the coordination of the overall Technical Report and for authoring Sections 1, 2, 3, 4, 5, 6, 15, 19, 21, 22, 23, 24, 25, 26, 27 and co-authoring Sections 16 and 18.
- I make regular visits to the Sugar Zone Project in my role as an employee of the Company.
- I am not independent of the Company applying the test set out in Section 1.5 of NI 43-101.
- I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F.

Dated this 5th day of March 2021 at Toronto, Ontario.

Original Signed and Sealed

Chris McCann, P.Eng.
Director, Technical Services
Harte Gold Corp.

28.2 Vincent Cardin-Tremblay, P.Geo.

I, Vincent Cardin-Tremblay, P.Geo., as an Author of this report titled *National Instrument (NI) 43-101 Technical Report—Expansion to 1,200 t/d for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario, Canada* (the Technical Report), with an Effective Date of 30 September 2020 for the Mineral Resource estimate, and 31 December 2020 for the updated Mineral Reserves estimate, and dated 5 March 2021, do hereby certify that:

- I am the Former Vice President, Geological Services of Harte Gold, with an office at Suite 2400 – 161 Bay St, Toronto, Ontario, M5J 2S1.
- I graduated from Université du Québec à Montréal, Québec in 2008 with a bachelor's degree in Geology.
- I am a member in good standing of the Professional Geoscientists of Ontario (Licence #3347) and of the Order of Geologists of Québec (License # 01386). My relevant experience is 13 years of experience as a geologist.
- I am a "Qualified Person" as that term is defined in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).
- I am responsible for authoring Sections 7, 8, 9, 10, 11, 12, 14.
- I have visited the Sugar Zone Project multiple times during my tenure with Harte Gold.
- I am not independent of the Company applying the test set out in Section 1.5 of NI 43-101.
- I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F.

Dated this 5th day of March, 2021 at Toronto, Ontario.

Original Signed and Sealed

Vincent Cardin-Tremblay, P.Geo.
Former Vice President, Geological Services
Harte Gold Corp.

28.3 Justin Taylor, P.Eng.

I, Justin Taylor, P.Eng., as an Author of this report titled *National Instrument (NI) 43-101 Technical Report—Expansion to 1,200 t/d for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario, Canada* (the Technical Report), with an Effective Date of 30 September 2020 for the Mineral Resource estimate, and 31 December 2020 for the updated Mineral Reserves estimate, and dated 5 March 2021, do hereby certify that:

- I am the President of Halyard Inc., with an office at 212 King St. W, Suite 501, Toronto, Ontario, M5H 1K5, Canada.
- I graduated from University of Pretoria in South Africa in 1999 with a Bachelor's Degree in Mechanical Engineering, Maintenance engineering in 2001 and a business diploma in 2004.
- I am a member in good standing of the Professional Engineers Ontario, License #100140330, Professional Engineers & Geoscientists Newfoundland & Labrador, License #06347 and Professional Engineers and Professional Geoscientists in the Northwest Territories and Nunavut, License #L4100.
- I am a "Qualified Person" as that term is defined in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).
- I am responsible for authoring Section 17 and co-authoring Section 13.
- I have made make regular visits to the Sugar Zone Project since 2016 and my last site visit was in October 2020.
- I am independent of the Company applying the test set out in Section 1.5 of NI 43-101.
- I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F.

Dated this 5th day of March, 2021 at Toronto, Ontario.

Original Signed and Sealed

Justin Taylor, P.Eng.
Principal, President, and Director
Halyard Inc.

28.4 Ivana Sabaj Abumohor, P.Eng.

I, Ivana Sabaj Abumohor, P.Eng., as an Author of this report titled *National Instrument (NI) 43-101 Technical Report—Expansion to 1,200 t/d for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario, Canada* (the Technical Report), with an Effective Date of 30 September 2020 for the Mineral Resource estimate, and 31 December 2020 for the updated Mineral Reserves estimate, and dated 5 March 2021, do hereby certify that:

- I am the Lead Process Engineer, Halyard Inc., with an office at 212 King St. W, Suite 501, Toronto, Ontario, M5H 1K5, Canada.
- I graduated from Universidad de Chile, Diagonal Paraguay 265, Santiago, Chile, in 2010 with a master's degree in Chemical Engineering.
- I am a member in good standing of the Professional Engineers Ontario, License #100528863. My relevant experience is 10 years as a Chemical Engineer.
- I am a "Qualified Person" as that term is defined in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).
- I am responsible for authoring Section 13 and co-authoring Section 17.
- I have made several visits to the Sugar Zone Project since 2018 and my last site visit was in February 2019.
- I am independent of the Company applying the test set out in Section 1.5 of NI 43-101.
- I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F.

Dated this 5th day of March, 2021 at Toronto, Ontario.

Original Signed and Sealed

Ivana Sabaj Abumohor, P.Eng.
Lead Process Engineer
Halyard Inc.

28.5 Rob Brown, P.Eng.

I, Rob Brown, P.Eng., as an Author of this report titled *National Instrument (NI) 43-101 Technical Report—Expansion to 1,200 t/d for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario, Canada* (the Technical Report), with an Effective Date of 30 September 2020 for the Mineral Resource estimate, and 31 December 2020 for the updated Mineral Reserves estimate, and dated 5 March 2021, do hereby certify that:

- I am the Managing Director, Canada, Paterson & Cooke Canada Inc., with an office at 1351C Kelly Lake Road, Unit #2, Sudbury, Ontario, Canada, P3E 5P5.
- I graduated from Laurentian University, Sudbury, Ontario in 1994 with a bachelor's degree in Mine Engineering.
- I am a member in good standing of the Professional Engineers Ontario, License #90499682. My relevant experience is 24 years in the mining industry, specializing in paste backfill.
- I am a "Qualified Person" as that term is defined in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).
- I am responsible for co-authoring Section 16.16.
- I have not visited the Sugar Zone Project.
- I am independent of the Company applying the test set out in Section 1.5 of NI 43-101.
- I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F.

Dated this 5th day of March 2021 at Ontario, Canada.

Original Signed and Sealed

Rob Brown, P.Eng.
Managing Director, Canada
Paterson & Cooke Canada Inc.

28.6 Ganan Nadarajah, P.Eng.

I, Ganan Nadarajah, P.Eng., as an Author of this report titled *National Instrument (NI) 43-101 Technical Report—Expansion to 1,200 t/d for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario, Canada* (the Technical Report), with an Effective Date of 30 September 2020 for the Mineral Resource estimate, and 31 December 2020 for the updated Mineral Reserves estimate, and dated 5 March 2021, do hereby certify that:

- I am a Senior Associate Geotechnical Engineer, Wood Canada Limited, with an office at 2020 Winston Park Drive, Suite 600, Oakville, Ontario, Canada, L6H 6X7.
- I graduated from the University of Peradeniya, Peradeniya, Sri Lanka in 1998 with a Bachelor's Degree in Civil Engineering and of University of British, Vancouver, BC, Canada in 2002 with a Master of Applied Science Degree in Geotechnical Engineering.
- I am a member in good standing of the Professional Engineers Ontario (License #100183764), and Association of Professional Engineers and Geoscientists of British Columbia (License #33280). My relevant experience is 18 years as a geotechnical engineer.
- I am a "Qualified Person" as that term is defined in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).
- I am responsible for authoring Sections 18.2.4, 18.3, and 26.1.6, and I am also responsible for relevant content in Sections 1.15.2, 2, 25 and 27 of the Technical Report.
- I have visited the Sugar Zone Project three times, most recently between 17 August and 18 August 2020.
- I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101, I have been involved in the past five years with performing technical and design works related to the tailings management facility at the Sugar Zone Project site. I am the Engineer of Record for the existing North Tailings Management Facility.
- I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F.

Dated this 5th day of March 2021 at Oakville, Ontario.

Original Signed and Sealed

Ganan Nadarajah, P.Eng.
Senior Associate Geotechnical Engineer
Wood Canada Limited

28.7 Ron C. deGagne, P.Geo.

I Ron C. deGagne, P.Geo., as an Author of this report titled *National Instrument (NI) 43-101 Technical Report—Expansion to 1,200 t/d for the Sugar Zone Mine, Sault Ste. Marie Mining Division, Ontario, Canada* (the Technical Report), with an Effective Date of 30 September 2020 for the Mineral Resource estimate, and 31 December 2020 for the updated Mineral Reserves estimate, and dated 5 March 2021, do hereby certify that:

- I am a Senior Geoscientist, at Environmental Applications Group (EAG) Inc., with an office at 2565 Kingsway, Unit 2, Sudbury, Ontario, P3B 2G1.
- I am a graduate of Sir Sandford Fleming College, Lindsay, Ontario in 1981 with a technology diploma in Earth Science.
- I am a member in good standing of the Professional Geoscientists of Ontario, License No. 0557. My relevant experience is 25 years as an independent consultant.
- I am a "Qualified Person" as that term is defined in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).
- I am responsible for authoring Section 20.
- I have not visited the Sugar Zone Project.
- I am Independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
- I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F.

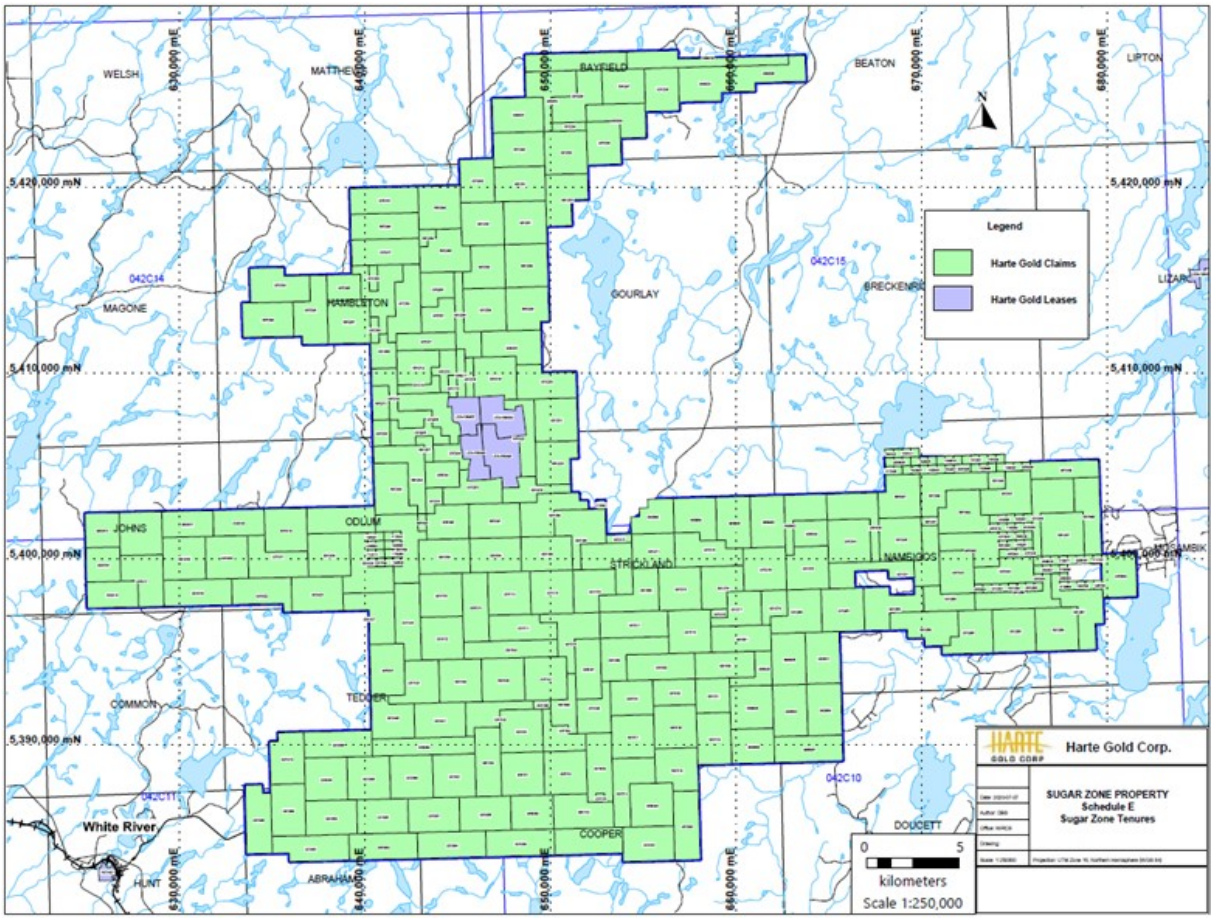
Dated this 5th day of March 2021 at Sudbury, Ontario.

Original Signed and Sealed

Ron C. deGange, P.Geo.
Senior Geoscientist
Environmental Applications Group Inc.

APPENDIX A

List of Claims and Mining Leases


WELSH
MATTHEWS
BAYFIELD
BEATON
LIPTON
630,000 mE
640,000 mE
650,000 mE
660,000 mE
670,000 mE
680,000 mE
5,420,000 mN
5,410,000 mN
5,400,000 mN
5,390,000 mN
042C14
042C15
042C10
042C11
MAGONE
HAMBLETON
GOURLAY
BRECKENRIDGE
LIZARD
JOHNS
ODLUM
STRICKLAND
NAMEGOS
COMMON
TEDDER
DOUCETT
COOPER
ABRAHAM
HUNT
White River
N
Legend
Harte Gold Claims
Harte Gold Leases
Harte Gold Corp.
SUGAR ZONE PROPERTY
Schedule E
Sugar Zone Tenures
0
5
kilometers
Scale 1:250,000

SCHEDULE "A" – HARTE GOLD CORP. – SUGAR ZONE CLAIMS

Legacy Claim Id	Township / Area	Tenure ID	Tenure Type	Anniversary Date	Work Required	Total Reserve
4281896	ODLUM	136581*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	334503*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	255919*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	237877*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	220822*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	220821*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	209284*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	209282*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	201257*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	171296*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	142560*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	136582*	Boundary Cell Mining Claim	2/6/2021	$200	$0
4281896	ODLUM	324599*	Single Cell Mining Claim	2/6/2021	$400	$0
4281896	ODLUM	255918*	Single Cell Mining Claim	2/6/2021	$400	$0
4281896	ODLUM	255917*	Single Cell Mining Claim	2/6/2021	$400	$223
4281896	ODLUM	209283*	Single Cell Mining Claim	2/6/2021	$400	$0
	MOSAMBIK	532869	Multi-cell Mining Claim	4/10/2021	8000	0
	NAMEIGOS	531281	Multi-cell Mining Claim	4/10/2021	10000	0
	NAMEIGOS	531282	Multi-cell Mining Claim	4/10/2021	9600	1753
	NAMEIGOS	531289	Multi-cell Mining Claim	4/10/2021	5600	2238
	NAMEIGOS	531331	Multi-cell Mining Claim	4/10/2021	7600	2016
	NAMEIGOS,STRICKLAND	531280	Multi-cell Mining Claim	4/10/2021	9600	0
	NAMEIGOS	514033	Single Cell Mining Claim	4/10/2021	400	0
	NAMEIGOS	514035	Single Cell Mining Claim	4/10/2021	400	0
	COOPER,STRICKLAND	531165	Multi-cell Mining Claim	4/10/2021	5200	1331
	HAMBLETON	531227	Multi-cell Mining Claim	4/10/2021	5600	1553
	HAMBLETON	531248	Multi-cell Mining Claim	4/10/2021	10000	0
	HAMBLETON	531265	Multi-cell Mining Claim	4/10/2021	10000	0
	HAMBLETON	531266	Multi-cell Mining Claim	4/10/2021	5600	0
	HAMBLETON	531267	Multi-cell Mining Claim	4/10/2021	5600	0
	ODLUM	531183	Multi-cell Mining Claim	4/10/2021	9600	1370
	ODLUM	531198	Multi-cell Mining Claim	4/10/2021	7600	3217
	ODLUM,STRICKLAND	531184	Multi-cell Mining Claim	4/10/2021	9600	2087
	ODLUM,STRICKLAND	531197	Multi-cell Mining Claim	4/10/2021	9600	3658
	ODLUM,STRICKLAND,TEDDER	531175	Multi-cell Mining Claim	4/10/2021	10000	187
	STRICKLAND	531157	Multi-cell Mining Claim	4/10/2021	10000	5781
	STRICKLAND,TEDDER	531169	Multi-cell Mining Claim	4/10/2021	8800	5224
	STRICKLAND,TEDDER	531171	Multi-cell Mining Claim	4/10/2021	8800	4401
	HAMBLETON	531254	Multi-cell Mining Claim	6/13/2021	9600	0
	HAMBLETON	531255	Multi-cell Mining Claim	6/13/2021	10000	0
	HAMBLETON	531256	Multi-cell Mining Claim	6/13/2021	10000	583
	HAMBLETON	531258	Multi-cell Mining Claim	6/13/2021	4800	0
	HAMBLETON	531269	Multi-cell Mining Claim	6/13/2021	1200	0
	NAMEIGOS	531335	Multi-cell Mining Claim	6/13/2021	10000	0
	NAMEIGOS	531340	Multi-cell Mining Claim	6/13/2021	6800	33
	NAMEIGOS	531342	Multi-cell Mining Claim	6/13/2021	8000	0
	NAMEIGOS	531343	Multi-cell Mining Claim	6/13/2021	8000	0
	NAMEIGOS	531344	Multi-cell Mining Claim	6/13/2021	7200	2174
4260661	ODLUM	205218	Boundary Cell Mining Claim	6/20/2021	200	0
4260665	ODLUM	236538	Boundary Cell Mining Claim	6/20/2021	200	837
4284301	ODLUM	113014	Boundary Cell Mining Claim	6/20/2021	200	374
4284301	ODLUM	323310	Boundary Cell Mining Claim	6/20/2021	200	832
	JOHNS	530313	Multi-cell Mining Claim	6/20/2021	6400	2174
	JOHNS	530314	Multi-cell Mining Claim	6/20/2021	6400	940
	JOHNS	530315	Multi-cell Mining Claim	6/20/2021	7200	4533
	JOHNS	530316	Multi-cell Mining Claim	6/20/2021	10000	0
	JOHNS	530317	Multi-cell Mining Claim	6/20/2021	7200	0
	JOHNS	531017	Multi-cell Mining Claim	6/20/2021	9600	5604
	JOHNS	531018	Multi-cell Mining Claim	6/20/2021	10000	0
	JOHNS,ODLUM	530318	Multi-cell Mining Claim	6/20/2021	7200	0
	JOHNS,ODLUM	531019	Multi-cell Mining Claim	6/20/2021	9600	0
	JOHNS,ODLUM	531020	Multi-cell Mining Claim	6/20/2021	10000	0
	ODLUM	531016	Multi-cell Mining Claim	6/20/2021	10000	0

	ODLUM	531021	Multi-cell Mining Claim	6/20/2021	10000	455
	ODLUM	531024	Multi-cell Mining Claim	6/20/2021	10000	0
	ODLUM	531025	Multi-cell Mining Claim	6/20/2021	9600	0
	ODLUM,TEDDER	531022	Multi-cell Mining Claim	6/20/2021	8800	247
	ODLUM,TEDDER	531023	Multi-cell Mining Claim	6/20/2021	9600	89
	ODLUM	531201	Multi-cell Mining Claim	10/29/2021	2000	398
	STRICKLAND	531162	Multi-cell Mining Claim	2020-11-16	9600	0
	STRICKLAND	531168	Multi-cell Mining Claim	2020-11-16	10000	0
	STRICKLAND	531177	Multi-cell Mining Claim	2020-11-16	9600	0
	STRICKLAND	531178	Multi-cell Mining Claim	2020-11-16	10000	0
	STRICKLAND	531180	Multi-cell Mining Claim	2020-11-16	9200	0
	STRICKLAND	531271	Multi-cell Mining Claim	2020-11-16	8000	0
	STRICKLAND	531273	Multi-cell Mining Claim	2020-11-16	10000	0
	STRICKLAND	531274	Multi-cell Mining Claim	2020-11-16	10000	0
	STRICKLAND	531275	Multi-cell Mining Claim	2020-11-16	8400	2439
	STRICKLAND	531278	Multi-cell Mining Claim	2020-11-16	800	0
	GOURLAY	531220	Multi-cell Mining Claim	2020-12-03	9600	0
	GOURLAY	531225	Multi-cell Mining Claim	2020-12-03	9600	0
	GOURLAY	531229	Multi-cell Mining Claim	2020-12-03	10000	0
	GOURLAY	531231	Multi-cell Mining Claim	2020-12-03	10000	0
	GOURLAY,HAMBLETON	531224	Multi-cell Mining Claim	2020-12-03	9600	0
	GOURLAY,HAMBLETON	531226	Multi-cell Mining Claim	2020-12-03	10000	0
	GOURLAY,HAMBLETON	531230	Multi-cell Mining Claim	2020-12-03	8800	0
	GOURLAY,HAMBLETON	531243	Multi-cell Mining Claim	2020-12-03	10000	0
	GOURLAY,HAMBLETON,STRICKLAND	531222	Multi-cell Mining Claim	2020-12-03	6200	0
	GOURLAY,STRICKLAND	531221	Multi-cell Mining Claim	2020-12-03	10000	0
	HAMBLETON	531228	Multi-cell Mining Claim	2020-12-03	6000	0
	ODLUM,STRICKLAND	531270	Multi-cell Mining Claim	2020-12-03	5000	0
	STRICKLAND	531167	Multi-cell Mining Claim	2020-12-03	8400	0
	STRICKLAND	531170	Multi-cell Mining Claim	2020-12-03	9200	0
	STRICKLAND	531176	Multi-cell Mining Claim	2020-12-03	10000	0
	STRICKLAND	531179	Multi-cell Mining Claim	2020-12-03	8400	0
	STRICKLAND	531181	Multi-cell Mining Claim	2020-12-03	9600	0
	STRICKLAND	531185	Multi-cell Mining Claim	2020-12-03	9600	0
	STRICKLAND	531195	Multi-cell Mining Claim	2020-12-03	8800	0
	STRICKLAND	531196	Multi-cell Mining Claim	2020-12-03	8800	0
	STRICKLAND	531223	Multi-cell Mining Claim	2020-12-03	7400	0
	STRICKLAND	531272	Multi-cell Mining Claim	2020-12-03	1200	0
4260617	STRICKLAND	110507	Single Cell Mining Claim	2020-12-03	200	0
	BAYFIELD,HAMBLETON,MATTHEWS	531242	Multi-cell Mining Claim	2020-12-17	8000	0
	GOURLAY,HAMBLETON	531241	Multi-cell Mining Claim	2020-12-17	9600	0
	HAMBLETON	531244	Multi-cell Mining Claim	2020-12-17	10000	0
	HAMBLETON	531245	Multi-cell Mining Claim	2020-12-17	9600	0
	HAMBLETON	531246	Multi-cell Mining Claim	2020-12-17	9600	0
	HAMBLETON	531247	Multi-cell Mining Claim	2020-12-17	9600	0
	HAMBLETON	531264	Multi-cell Mining Claim	2020-12-17	9600	0
	BAYFIELD	531235	Multi-cell Mining Claim	2020-12-22	8000	0
	BAYFIELD	531236	Multi-cell Mining Claim	2020-12-22	8000	0
	BAYFIELD	531237	Multi-cell Mining Claim	2020-12-22	8000	0
	BAYFIELD	531238	Multi-cell Mining Claim	2020-12-22	9200	0
	BAYFIELD	531239	Multi-cell Mining Claim	2020-12-22	1600	0
	BAYFIELD,GOURLAY	531233	Multi-cell Mining Claim	2020-12-22	10000	0
	BAYFIELD,GOURLAY	531234	Multi-cell Mining Claim	2020-12-22	8000	0
	BAYFIELD,GOURLAY,HAMBLETON	531240	Multi-cell Mining Claim	2020-12-22	9600	0
	GOURLAY	531232	Multi-cell Mining Claim	2020-12-22	9600	0
4260661	ODLUM	137166	Boundary Cell Mining Claim	2020-12-23	200	930
4260661	ODLUM	156716	Boundary Cell Mining Claim	2020-12-23	200	548
4260661	ODLUM	142645	Boundary Cell Mining Claim	2020-12-23	200	151
4260664	ODLUM	308490	Boundary Cell Mining Claim	2020-12-23	200	111
4260664	ODLUM	168606	Boundary Cell Mining Claim	2020-12-23	200	174
4260665	ODLUM	112652	Boundary Cell Mining Claim	2020-12-23	200	0
4260665	ODLUM	199956	Boundary Cell Mining Claim	2020-12-23	200	298
4260665	ODLUM	155301	Boundary Cell Mining Claim	2020-12-23	200	236
	HAMBLETON	531210	Multi-cell Mining Claim	2020-12-23	6800	6082

	HAMBLETON	531249	Multi-cell Mining Claim	2020-12-23	1200	0
	HAMBLETON	531257	Multi-cell Mining Claim	2020-12-23	10000	0
	HAMBLETON	531268	Multi-cell Mining Claim	2020-12-23	4000	0
	HAMBLETON,ODLUM	531209	Multi-cell Mining Claim	2020-12-23	2400	1604
	ODLUM	531026	Multi-cell Mining Claim	2020-12-23	10000	0
	ODLUM	531182	Multi-cell Mining Claim	2020-12-23	10000	0
	ODLUM	531199	Multi-cell Mining Claim	2020-12-23	800	0
	ODLUM	531200	Multi-cell Mining Claim	2020-12-23	10000	0
	ODLUM,TEDDER	531027	Multi-cell Mining Claim	2020-12-23	9600	0
	ODLUM,TEDDER	531154	Multi-cell Mining Claim	2020-12-23	10000	0
	ODLUM,TEDDER	531173	Multi-cell Mining Claim	2020-12-23	10000	0
	ODLUM,TEDDER	531174	Multi-cell Mining Claim	2020-12-23	9600	0
	STRICKLAND,TEDDER	531156	Multi-cell Mining Claim	2020-12-23	10000	0
	TEDDER	531031	Multi-cell Mining Claim	2020-12-23	9600	0
	TEDDER	531153	Multi-cell Mining Claim	2020-12-23	8800	0
	TEDDER	531155	Multi-cell Mining Claim	2020-12-23	10000	0
	TEDDER	531172	Multi-cell Mining Claim	2020-12-23	10000	0
	ODLUM	531203	Multi-cell Mining Claim	2020-12-31	7000	0
	ODLUM	531204	Multi-cell Mining Claim	2020-12-31	3800	0
4288587	NAMEIGOS	125769	Boundary Cell Mining Claim	2021-01-08	200	0
4288587	NAMEIGOS	286343	Boundary Cell Mining Claim	2021-01-08	200	0
4288587	NAMEIGOS	286342	Boundary Cell Mining Claim	2021-01-08	200	0
4288587	NAMEIGOS	286341	Boundary Cell Mining Claim	2021-01-08	200	0
4288587	NAMEIGOS	274252	Boundary Cell Mining Claim	2021-01-08	200	0
4288587	NAMEIGOS	266283	Boundary Cell Mining Claim	2021-01-08	200	0
4288587	NAMEIGOS	189153	Boundary Cell Mining Claim	2021-01-08	200	11
4288587	NAMEIGOS	170388	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	102955	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	322925	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	286384	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	227074	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	219128	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	189186	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	170921	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	125817	Boundary Cell Mining Claim	2021-01-08	200	149
4288588	NAMEIGOS	102957	Boundary Cell Mining Claim	2021-01-08	200	0
4288588	NAMEIGOS	102956	Boundary Cell Mining Claim	2021-01-08	200	0
4288589	NAMEIGOS	287639	Boundary Cell Mining Claim	2021-01-08	200	0
4288589	NAMEIGOS	267591	Boundary Cell Mining Claim	2021-01-08	200	0
4288589	NAMEIGOS	220366	Boundary Cell Mining Claim	2021-01-08	200	423
4288589	NAMEIGOS	208950	Boundary Cell Mining Claim	2021-01-08	200	0
4288589	NAMEIGOS	173870	Boundary Cell Mining Claim	2021-01-08	200	0
4288589	NAMEIGOS	155027	Boundary Cell Mining Claim	2021-01-08	200	0
4288589	NAMEIGOS	117345	Boundary Cell Mining Claim	2021-01-08	200	0
4288589	NAMEIGOS	335993	Single Cell Mining Claim	2021-01-08	400	0
4288589	NAMEIGOS	220373	Single Cell Mining Claim	2021-01-08	400	423
4288589	NAMEIGOS	208958	Single Cell Mining Claim	2021-01-08	400	0
4288231	NAMEIGOS	104062	Boundary Cell Mining Claim	2021-01-09	200	0
4288231	NAMEIGOS	225048	Boundary Cell Mining Claim	2021-01-09	200	0
4288231	NAMEIGOS	159665	Boundary Cell Mining Claim	2021-01-09	200	0
	ABRAHAM,COOPER,TEDDER	531096	Multi-cell Mining Claim	2021-01-09	10000	0
	ABRAHAM,TEDDER	531094	Multi-cell Mining Claim	2021-01-09	10000	0
	ABRAHAM,TEDDER	531095	Multi-cell Mining Claim	2021-01-09	10000	0
	COOPER	531112	Multi-cell Mining Claim	2021-01-09	10000	0
	COOPER	531139	Multi-cell Mining Claim	2021-01-09	9200	0
	COOPER	531163	Multi-cell Mining Claim	2021-01-09	6000	0
	COOPER,STRICKLAND	531166	Multi-cell Mining Claim	2021-01-09	800	0
	COOPER,STRICKLAND,TEDDER	531152	Multi-cell Mining Claim	2021-01-09	6800	0
	COOPER,TEDDER	531097	Multi-cell Mining Claim	2021-01-09	10000	0
	COOPER,TEDDER	531100	Multi-cell Mining Claim	2021-01-09	9600	0
	COOPER,TEDDER	531111	Multi-cell Mining Claim	2021-01-09	10000	0
	COOPER,TEDDER	531151	Multi-cell Mining Claim	2021-01-09	10000	0
	MOSAMBIK	531287	Multi-cell Mining Claim	2021-01-09	10000	0
	MOSAMBIK	531348	Multi-cell Mining Claim	2021-01-09	8800	0

	MOSAMBIK, NAMEIGOS	531286	Multi-cell Mining Claim	2021-01-09	10000	0
	MOSAMBIK, NAMEIGOS	531288	Multi-cell Mining Claim	2021-01-09	8400	0
	MOSAMBIK, NAMEIGOS	531347	Multi-cell Mining Claim	2021-01-09	10000	0
	MOSAMBIK, NAMEIGOS	531349	Multi-cell Mining Claim	2021-01-09	6400	0
	MOSAMBIK, NAMEIGOS	531350	Multi-cell Mining Claim	2021-01-09	10000	0
	NAMEIGOS	531283	Multi-cell Mining Claim	2021-01-09	10000	0
	NAMEIGOS	531284	Multi-cell Mining Claim	2021-01-09	9200	0
	NAMEIGOS	531285	Multi-cell Mining Claim	2021-01-09	10000	0
	NAMEIGOS	531351	Multi-cell Mining Claim	2021-01-09	9600	0
	NAMEIGOS	531352	Multi-cell Mining Claim	2021-01-09	10000	0
	TEDDER	531046	Multi-cell Mining Claim	2021-01-09	8800	0
	TEDDER	531047	Multi-cell Mining Claim	2021-01-09	9600	0
	TEDDER	531079	Multi-cell Mining Claim	2021-01-09	9200	0
	TEDDER	531098	Multi-cell Mining Claim	2021-01-09	9600	0
	TEDDER	531099	Multi-cell Mining Claim	2021-01-09	9600	0
	COOPER	531126	Single Cell Mining Claim	2021-01-09	400	0
04288250	MOSAMBIK	125756	Single Cell Mining Claim	2021-01-09	400	0
04288250	MOSAMBIK	293144	Single Cell Mining Claim	2021-01-09	400	0
04288250	MOSAMBIK	274244	Single Cell Mining Claim	2021-01-09	400	0
04288250	MOSAMBIK	273605	Single Cell Mining Claim	2021-01-09	400	0
04288250	MOSAMBIK	153728	Single Cell Mining Claim	2021-01-09	400	0
4288237	MOSAMBIK	118071	Single Cell Mining Claim	2021-01-09	400	0
4288237	MOSAMBIK	273604	Single Cell Mining Claim	2021-01-09	400	0
4288237	MOSAMBIK	226382	Single Cell Mining Claim	2021-01-09	400	0
4288237	MOSAMBIK	188477	Single Cell Mining Claim	2021-01-09	400	0
4288237	MOSAMBIK	170250	Single Cell Mining Claim	2021-01-09	400	0
4288249	MOSAMBIK	117527	Single Cell Mining Claim	2021-01-09	400	0
4288249	MOSAMBIK	336697	Single Cell Mining Claim	2021-01-09	400	0
4288249	MOSAMBIK	276267	Single Cell Mining Claim	2021-01-09	400	0
4288249	MOSAMBIK	221060	Single Cell Mining Claim	2021-01-09	400	0
4288237	MOSAMBIK, NAMEIGOS	344618	Single Cell Mining Claim	2021-01-09	400	0
4288237	MOSAMBIK, NAMEIGOS	265657	Single Cell Mining Claim	2021-01-09	400	0
4288230	NAMEIGOS	103256	Single Cell Mining Claim	2021-01-09	400	0
4288230	NAMEIGOS	127131	Single Cell Mining Claim	2021-01-09	400	0
4288232	NAMEIGOS	102261	Single Cell Mining Claim	2021-01-09	400	0
4288232	NAMEIGOS	276303	Single Cell Mining Claim	2021-01-09	400	0
4288232	NAMEIGOS	229063	Single Cell Mining Claim	2021-01-09	400	0
4288232	NAMEIGOS	219164	Single Cell Mining Claim	2021-01-09	400	0
4288232	NAMEIGOS	170953	Single Cell Mining Claim	2021-01-09	400	0
4288232	NAMEIGOS	118285	Single Cell Mining Claim	2021-01-09	400	0
4288233	NAMEIGOS	286410	Single Cell Mining Claim	2021-01-09	400	0
4288233	NAMEIGOS	189211	Single Cell Mining Claim	2021-01-09	400	0
4288233	NAMEIGOS	170954	Single Cell Mining Claim	2021-01-09	400	0
4288233	NAMEIGOS	154316	Single Cell Mining Claim	2021-01-09	400	0
4288233	NAMEIGOS	125852	Single Cell Mining Claim	2021-01-09	400	0
4288233	NAMEIGOS	118287	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531290	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531291	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531292	Single Cell Mining Claim	2021-01-09	400	0
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	NAMEIGOS	531301	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531302	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531304	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531305	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531306	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531309	Single Cell Mining Claim	2021-01-09	400	0
	NAMEIGOS	531316	Single Cell Mining Claim	2021-01-09	400	0

	NAMEIGOS	531317	Single Cell Mining Claim	2021-01-09	400	0
	COOPER	531115	Multi-cell Mining Claim	2021-01-10	9200	0
	COOPER	531116	Multi-cell Mining Claim	2021-01-10	9600	0
	COOPER	531117	Multi-cell Mining Claim	2021-01-10	10000	0
	COOPER	531118	Multi-cell Mining Claim	2021-01-10	10000	0
	COOPER,STRICKLAND	531119	Multi-cell Mining Claim	2021-01-10	8000	0
	COOPER,STRICKLAND	531120	Multi-cell Mining Claim	2021-01-10	6000	0
	COOPER,STRICKLAND	531121	Multi-cell Mining Claim	2021-01-10	6400	0
	COOPER,STRICKLAND	531164	Multi-cell Mining Claim	2021-01-10	7200	0
	ABRAHAM	531086	Multi-cell Mining Claim	2021-01-18	9600	0
	ABRAHAM,COOPER	531087	Multi-cell Mining Claim	2021-01-18	9600	0
4281802	NAMEIGOS	134919	Boundary Cell Mining Claim	2021-02-16	200	0
4281802	NAMEIGOS	302908	Boundary Cell Mining Claim	2021-02-16	200	0
4281802	NAMEIGOS	281507	Boundary Cell Mining Claim	2021-02-16	200	0
4281802	NAMEIGOS	151061	Boundary Cell Mining Claim	2021-02-16	200	0
4281802	NAMEIGOS	150356	Boundary Cell Mining Claim	2021-02-16	200	0
4281802	NAMEIGOS	141005	Boundary Cell Mining Claim	2021-02-16	200	1139
4281805	NAMEIGOS	122945	Boundary Cell Mining Claim	2021-02-16	200	0
4281805	NAMEIGOS	290157	Boundary Cell Mining Claim	2021-02-16	200	0
4281805	NAMEIGOS	186333	Boundary Cell Mining Claim	2021-02-16	200	0
4281805	NAMEIGOS	133689	Boundary Cell Mining Claim	2021-02-16	200	0
4285671	NAMEIGOS	186239	Boundary Cell Mining Claim	2021-02-16	200	0
4285671	NAMEIGOS	319552	Boundary Cell Mining Claim	2021-02-16	200	0
4285671	NAMEIGOS	282751	Boundary Cell Mining Claim	2021-02-16	200	0
4285671	NAMEIGOS	186240	Boundary Cell Mining Claim	2021-02-16	200	0
4285672	NAMEIGOS	157827	Boundary Cell Mining Claim	2021-02-16	200	0
4285672	NAMEIGOS	344511	Boundary Cell Mining Claim	2021-02-16	200	0
4285672	NAMEIGOS	238950	Boundary Cell Mining Claim	2021-02-16	200	0
	NAMEIGOS	531332	Multi-cell Mining Claim	2021-02-16	9600	768
	NAMEIGOS	531333	Multi-cell Mining Claim	2021-02-16	4800	0
	NAMEIGOS	531334	Multi-cell Mining Claim	2021-02-16	10000	0
	NAMEIGOS	531336	Multi-cell Mining Claim	2021-02-16	9200	0
	NAMEIGOS	531337	Multi-cell Mining Claim	2021-02-16	9200	0
	NAMEIGOS	531338	Multi-cell Mining Claim	2021-02-16	9600	0
	NAMEIGOS	531341	Multi-cell Mining Claim	2021-02-16	800	0
	NAMEIGOS	531345	Multi-cell Mining Claim	2021-02-16	800	0
	NAMEIGOS	531346	Multi-cell Mining Claim	2021-02-16	1600	496
	ABRAHAM	531081	Multi-cell Mining Claim	2021-02-22	10000	0
	ABRAHAM	531082	Multi-cell Mining Claim	2021-02-22	9600	0
	ABRAHAM	531083	Multi-cell Mining Claim	2021-02-22	9600	0
	ABRAHAM,TEDDER	531048	Multi-cell Mining Claim	2021-02-22	9000	859
	ABRAHAM,TEDDER	531080	Multi-cell Mining Claim	2021-02-22	9600	0
	NAMEIGOS,STRICKLAND	531276	Multi-cell Mining Claim	2021-02-22	10000	0
	NAMEIGOS,STRICKLAND	531279	Multi-cell Mining Claim	2021-02-22	4000	0
	STRICKLAND	531160	Multi-cell Mining Claim	2021-02-22	8400	0
	STRICKLAND	531161	Multi-cell Mining Claim	2021-02-22	8400	0
	STRICKLAND	531277	Multi-cell Mining Claim	2021-02-22	7200	0
	ABRAHAM,COOPER	531084	Multi-cell Mining Claim	2021-03-10	9600	0
	COOPER	531085	Multi-cell Mining Claim	2021-03-10	9600	0
	COOPER	531088	Multi-cell Mining Claim	2021-03-10	9600	0
	COOPER	531089	Multi-cell Mining Claim	2021-03-10	8000	0
	COOPER	531090	Multi-cell Mining Claim	2021-03-10	9600	0
	COOPER	531091	Multi-cell Mining Claim	2021-03-10	9600	0
	COOPER	531092	Multi-cell Mining Claim	2021-03-10	9600	0
	COOPER	531093	Multi-cell Mining Claim	2021-03-10	10000	0
	COOPER	531113	Multi-cell Mining Claim	2021-03-10	10000	0
	COOPER	531114	Multi-cell Mining Claim	2021-03-10	10000	0
	ODLUM	531205	Multi-cell Mining Claim	2021-03-27	4800	278
	HAMBLETON,ODLUM	531206	Multi-cell Mining Claim	2021-04-26	8200	345634
	BAYFIELD	549597	Multi-cell Mining Claim	2021-05-10	9600	0
	BAYFIELD	549623	Multi-cell Mining Claim	2021-05-10	9200	0
	BAYFIELD	549624	Multi-cell Mining Claim	2021-05-10	9600	0
	BAYFIELD	549625	Multi-cell Mining Claim	2021-05-10	8800	0
	BAYFIELD,BEATON	549626	Multi-cell Mining Claim	2021-05-10	9200	0

BAYFIELD,BEATON	549916	Multi-cell Mining Claim	2021-05-10	10000	0
ODLUM	531207	Multi-cell Mining Claim	2021-07-02	1600	36193
HAMBLETON	531214	Multi-cell Mining Claim	2021-07-20	2400	105705
GOURLAY,HAMBLETON	531219	Multi-cell Mining Claim	2021-11-20	9200	11993
HAMBLETON	531211	Multi-cell Mining Claim	2021-12-23	3200	2381
ODLUM	531202	Multi-cell Mining Claim	2021-12-23	9200	19310
HAMBLETON	531212	Multi-cell Mining Claim	2021-12-31	7200	47190
HAMBLETON	531215	Multi-cell Mining Claim	2021-12-31	3600	211070
HAMBLETON	531216	Multi-cell Mining Claim	2021-12-31	1000	467817
HAMBLETON	531217	Multi-cell Mining Claim	2021-12-31	2200	342089
HAMBLETON	531218	Multi-cell Mining Claim	2021-12-31	1800	126580
HAMBLETON,ODLUM	531208	Multi-cell Mining Claim	2021-12-31	5200	9687
HAMBLETON	531259	Multi-cell Mining Claim	2022-12-23	1200	851
COOPER	564960	Multi-cell Mining Claim	11/29/2021	Active	100
COOPER, DOUCETT	564961	Multi-cell Mining Claim	11/29/2021	Active	100
COOPER, DOUCETT, NAMEIGOS, STRICKLAND	564909	Multi-cell Mining Claim	11/29/2021	Active	100
COOPER, STRICKLAND	564959	Multi-cell Mining Claim	11/29/2021	Active	100
DOUCETT, NAMEIGOS	565900	Multi-cell Mining Claim	11/29/2021	Active	100
NAMEIGOS	564962	Multi-cell Mining Claim	11/29/2021	Active	100
NAMEIGOS	565901	Multi-cell Mining Claim	11/29/2021	Active	100
NAMEIGOS, STRICKLAND	564908	Multi-cell Mining Claim	11/29/2021	Active	100
NAMEIGOS, STRICKLAND	564963	Multi-cell Mining Claim	11/29/2021	Active	100
STRICKLAND	564958	Multi-cell Mining Claim	11/29/2021	Active	100
STRICKLAND	564964	Multi-cell Mining Claim	11/29/2021	Active	100
STRICKLAND	564965	Multi-cell Mining Claim	11/29/2021	Active	100
STRICKLAND	564966	Multi-cell Mining Claim	11/29/2021	Active	100

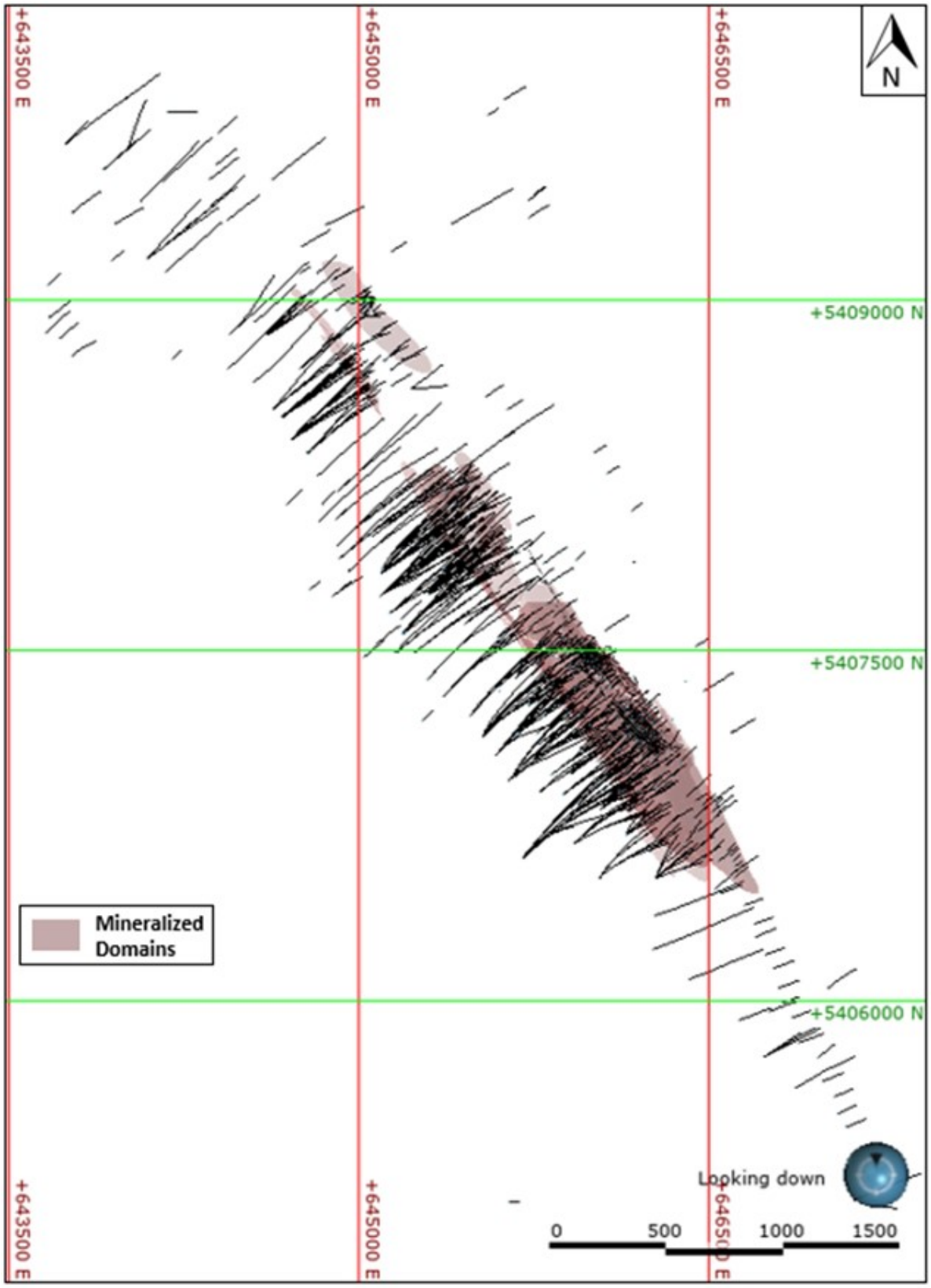
SCHEDULE "B" – HARTE GOLD CORP. – SUGAR ZONE LEASES

Claim #	Twp.	Issued	Anniversary	Area (Ha.)	Reserve		Lease #	Rights	PIN	Reg'd Plan
1069332	HAMBLETON	1-Jun-15	31-May-36	393.38	$38,629	LEA-109605	CLM514	MR+SR	31054-0003	Pts. 1-9, 1R-13011
1069333	HAMBLETON					LEA-109605	CLM514	MR+SR	31054-0004	
1069343	HAMBLETON					LEA-109605	CLM514	MR+SR	31054-0005	
1069344	HAMBLETON					LEA-109605	CLM514	MR+SR, MRO	31054-0006	
1069345	HAMBLETON					LEA-109605	CLM514	MR+SR, MRO		
1069346	HAMBLETON					LEA-109605	CLM514	MR+SR		
1182993	HAMBLETON					LEA-109605	CLM514	MR+SR		
1232640	GOURLAY					LEA-109605	CLM514	MR+SR, MRO		
1235595	HAMBLETON					LEA-109605	CLM514	MR+SR, MRO		
1069327	HAMBLETON	1-May-15	30-Apr-36	282.67	$3,347,763	LEA-109602	CLM515	MR+SR, MRO	31053-0001	Pts. 1-9, 1R-13039
1069328	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069329	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069330	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069331	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069334	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069335	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069336	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069337	HAMBLETON					LEA-109602	CLM515	MR+SR, MRO		
1069338	HAMBLETON					LEA-109602	CLM515	MR+SR, MRO		
1069339	HAMBLETON					LEA-109602	CLM515	MR+SR, MRO		
1069340	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069341	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069342	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069347	HAMBLETON					LEA-109602	CLM515	MR+SR		
1069348	HAMBLETON					LEA-109602	CLM515	MR+SR, MRO		
1069349	HAMBLETON					LEA-109602	CLM515	MR+SR, MRO		
1069350	HAMBLETON					LEA-109602	CLM515	MR+SR, MRO		
1135498	HAMBLETON					LEA-109602	CLM515	MR+SR		
1182994	HAMBLETON					LEA-109602	CLM515	MR+SR		
4270162	HAMBLETON					LEA-109602	CLM515	MR+SR		
937770	ODLUM	1-May-15	30-Apr-36	279.83	$690,651	LEA-109593	CLM516	MR+SR	31078-0001	Pts. 1-11, 1R-13038
1043803	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1043811	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1043812	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1069356	ODLUM					LEA-109593	CLM516	MR+SR		
1069357	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1069358	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1069363	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1069364	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1069365	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1069372	ODLUM					LEA-109593	CLM516	MRO		
1069373	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1069374	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1078250	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1078251	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1078252	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
1135499	HAMBLETON					LEA-109593	CLM516	MR+SR		
1194337	HAMBLETON					LEA-109593	CLM516	MR+SR		
1194340	ODLUM					LEA-109593	CLM516	MR+SR, MRO		
937771	ODLUM	1-May-15	30-Apr-36	511.38	$87,131	LEA-109592	CLM517	MR+SR	31077-0001	Pts. 1-8, 1R-13019
937772	ODLUM					LEA-109592	CLM517	MR+SR		
1043806	ODLUM					LEA-109592	CLM517	MR+SR, MRO		
1043807	ODLUM					LEA-109592	CLM517	MR+SR		
1043808	ODLUM					LEA-109592	CLM517	MR+SR, MRO		
1043809	ODLUM					LEA-109592	CLM517	MR+SR, MRO		
1043810	ODLUM					LEA-109592	CLM517	MRO		
1069352	HAMBLETON					LEA-109592	CLM517	MR+SR		
1069353	HAMBLETON					LEA-109592	CLM517	MR+SR, MRO		
1069354	ODLUM					LEA-109592	CLM517	MR+SR, MRO		
1069355	ODLUM					LEA-109592	CLM517	MR+SR		
1069366	ODLUM					LEA-109592	CLM517	MR+SR, MRO		
1069367	ODLUM					LEA-109592	CLM517	MR+SR, MRO		

1069368	ODLUM	LEA-109592	CLM517	MR+SR, MRO		
1069369	ODLUM	LEA-109592	CLM517	MR+SR, MRO		
1069370	ODLUM	LEA-109592	CLM517	MR+SR, MRO		
1069371	ODLUM	LEA-109592	CLM517	MR+SR, MRO		
1140638	STRICKLAND	LEA-109592	CLM517	MR+SR, MRO		
1140639	STRICKLAND	LEA-109592	CLM517	MR+SR, MRO		
1140640	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140641	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140642	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140643	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140644	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140645	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140646	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140647	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140658	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140659	STRICKLAND	LEA-109592	CLM517	MR+SR		
1140660	STRICKLAND	LEA-109592	CLM517	MR+SR		

APPENDIX B

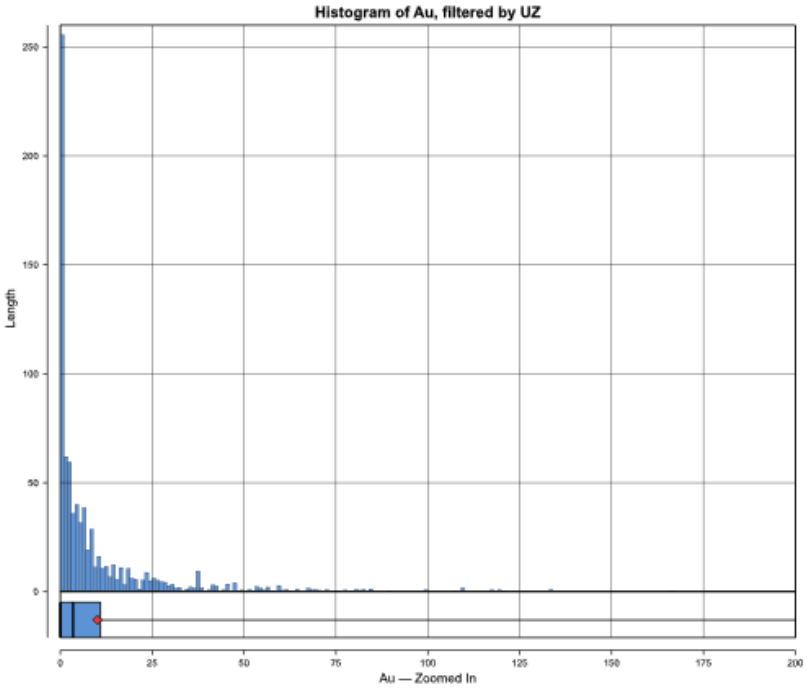
Surface Drill Hole Plan

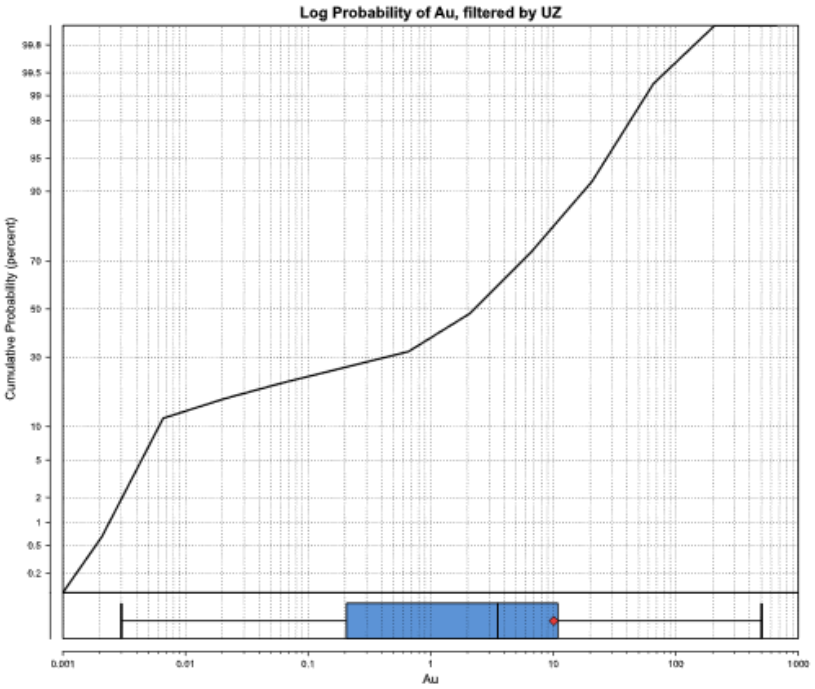

+643500 E
+645000 E
+646500 E
N
+5409000 N
+5407500 N
+5406000 N
Mineralized Domains
Looking down
0
500
1000
1500

APPENDIX C

Frequency Histograms and Probability Plots of Composites

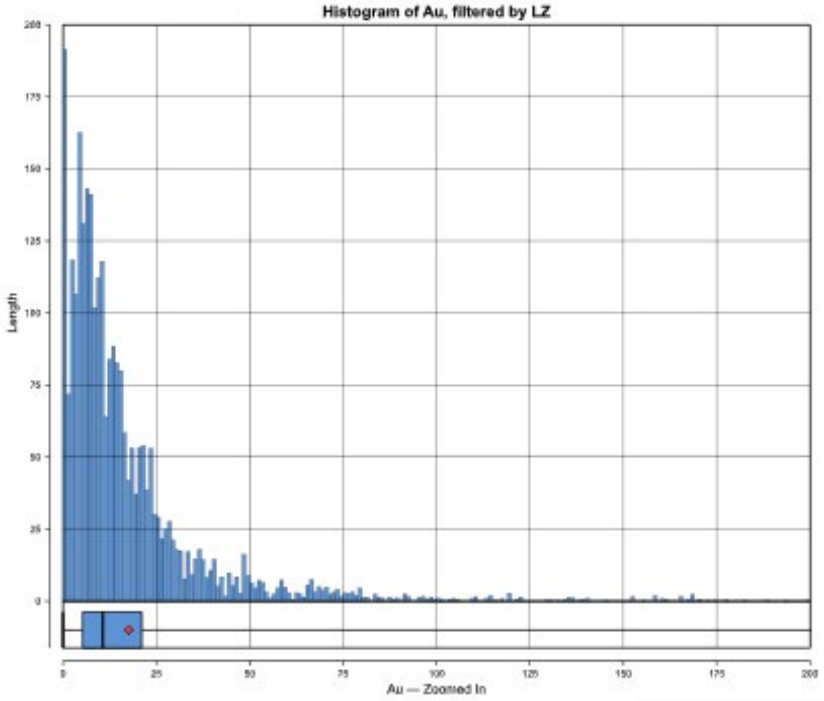
Sugar Zone Upper – UZ

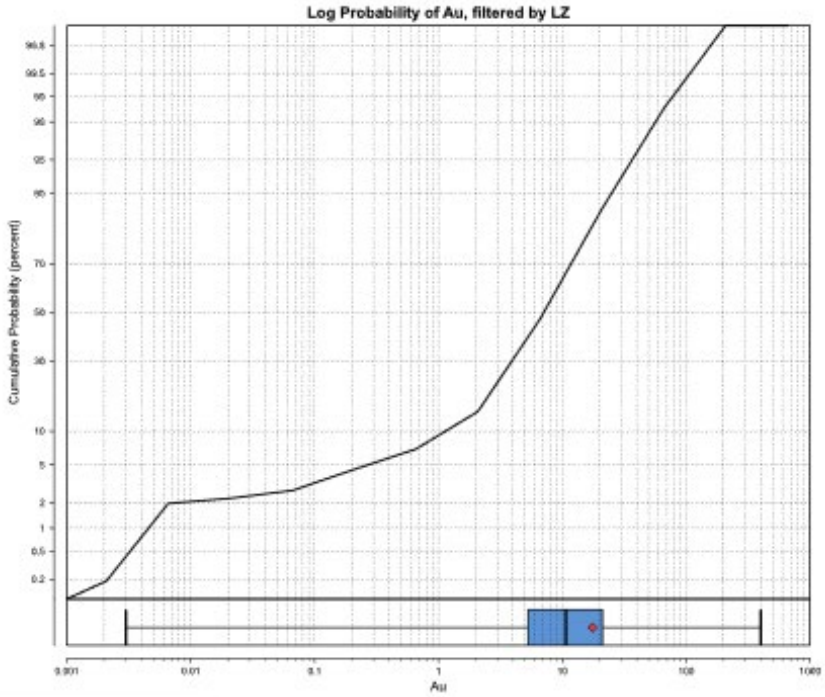




	Weighted Value
Count	850
Length	796.1
Mean	10.119
SD	18.958
CV	1.873
Variance	359.416
Minimum	0.003
Q1	0.206
Q2	3.521
Q3	10.916
Maximum	502.183

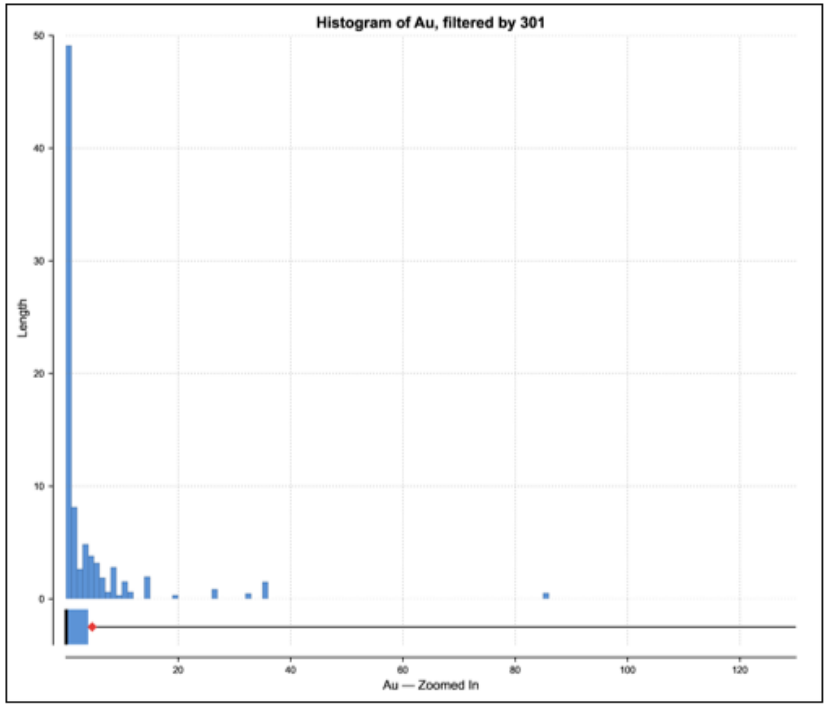
Sugar Zone Upper – LZ

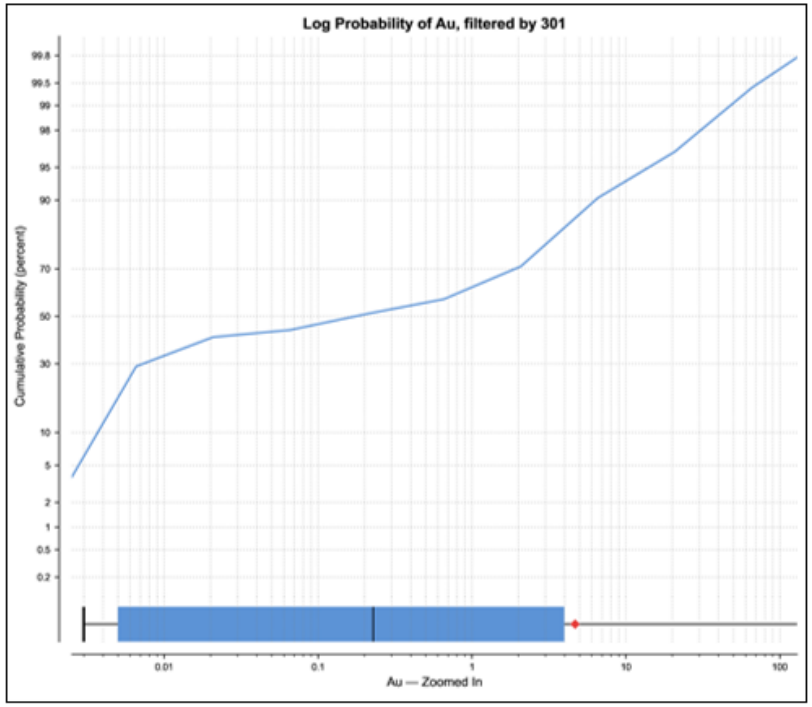




	Weighted Value
Count	2,460
Length	2,729.4
Mean	17.620
SD	23.768
CV	1.349
Variance	564.910
Minimum	0.003
Q1	5.277
Q2	10.721
Q3	21.113
Maximum	401.290

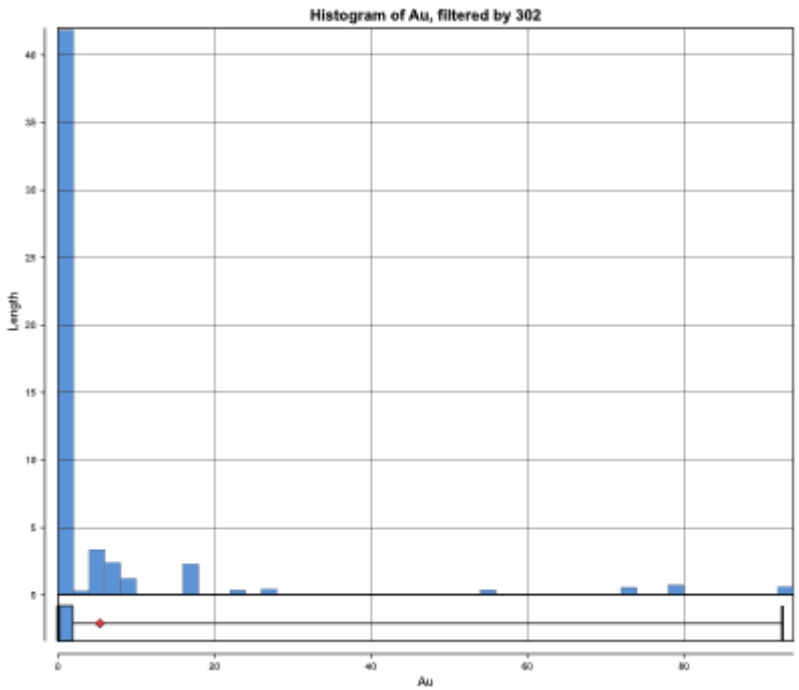
Sugar Zone FW1 - 301

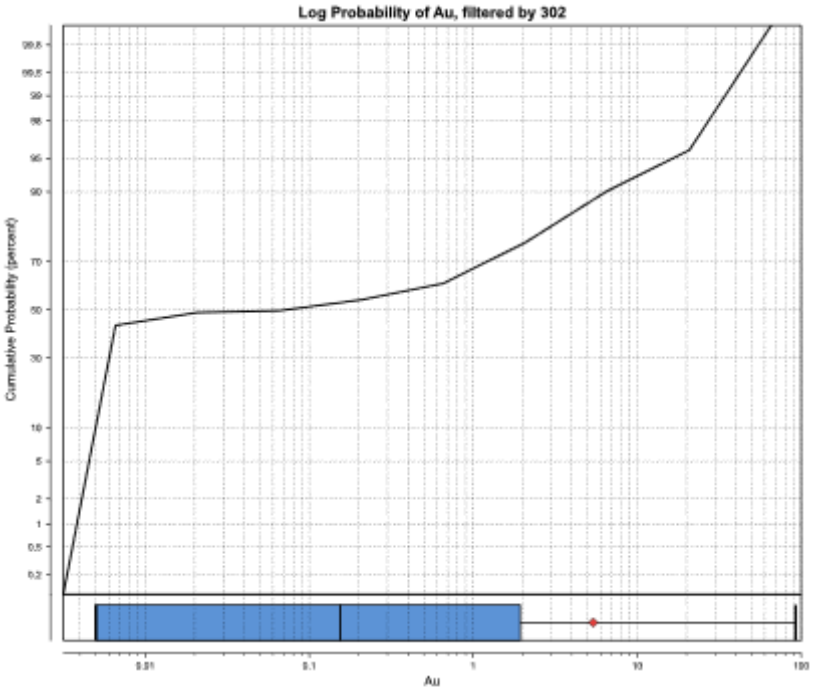




	Weighted Value
Count	118
Length	85.4
Mean	4.683
SD	16.219
CV	3.463
Variance	263.052
Minimum	0.003
Q1	0.005
Q2	0.228
Q3	3.984
Maximum	179.000

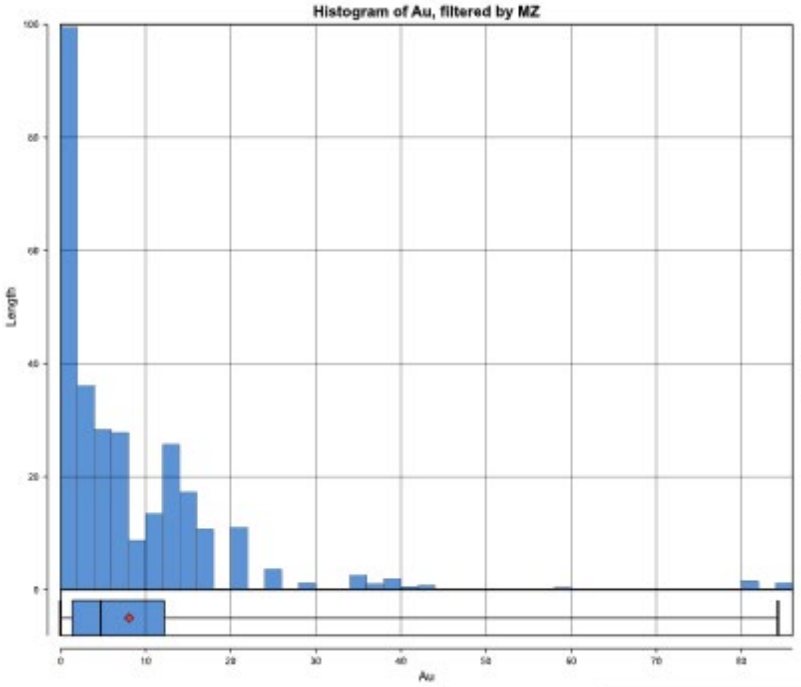
Sugar Zone FW2 – 302

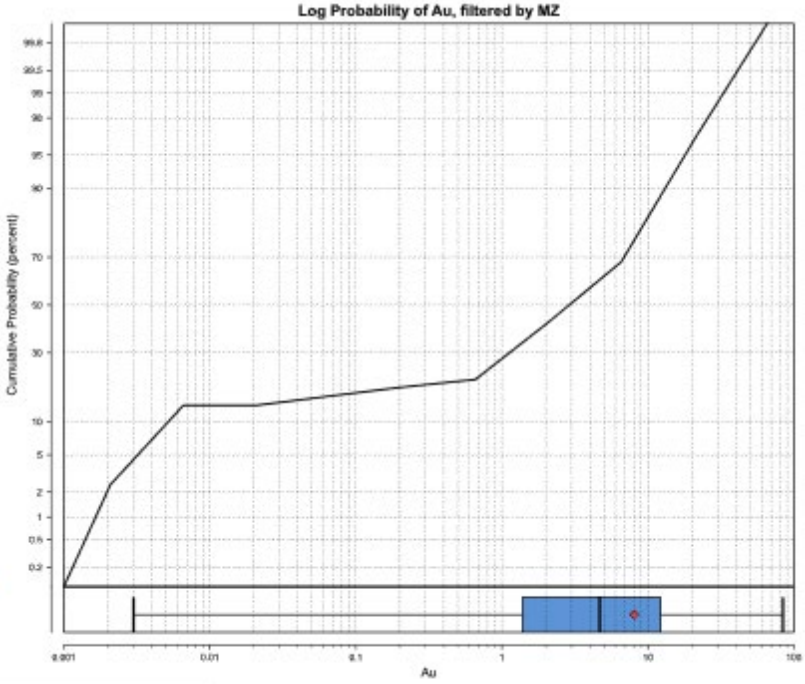




	Weighted Value
Count	79
Length	54.5
Mean	5.399
SD	15.938
CV	2.952
Variance	254.010
Minimum	0.005
Q1	0.005
Q2	0.154
Q3	1.934
Maximum	92.600

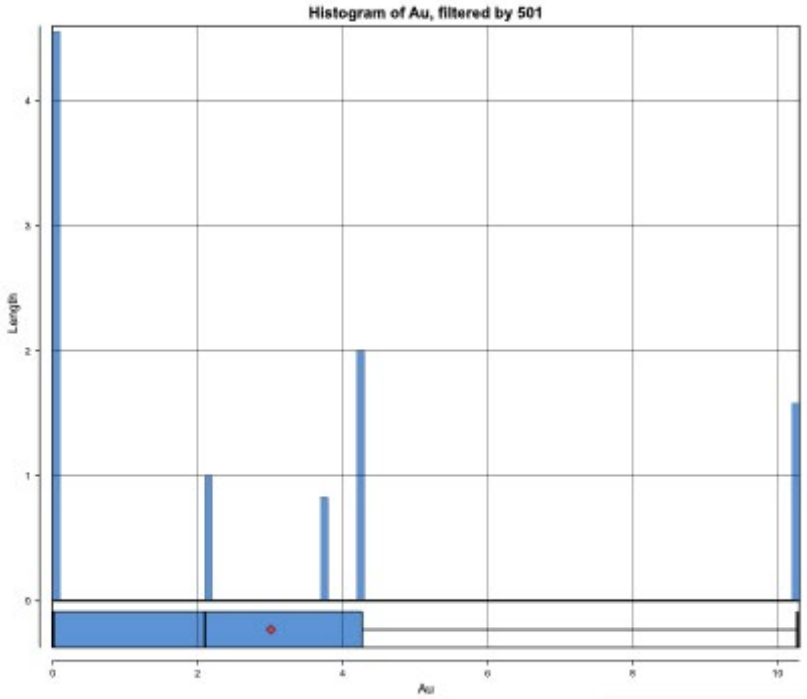
Middle Zone – MZ

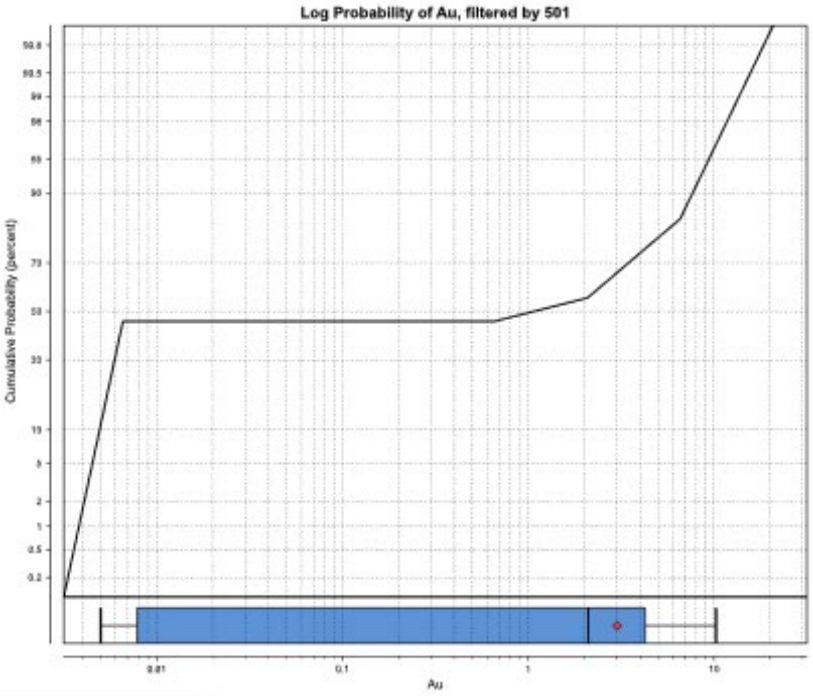




	Weighted Value
Count	149
Length	293.9
Mean	8.075
SD	11.079
CV	1.372
Variance	122.748
Minimum	0.003
Q1	1.381
Q2	4.680
Q3	12.198
Maximum	84.288

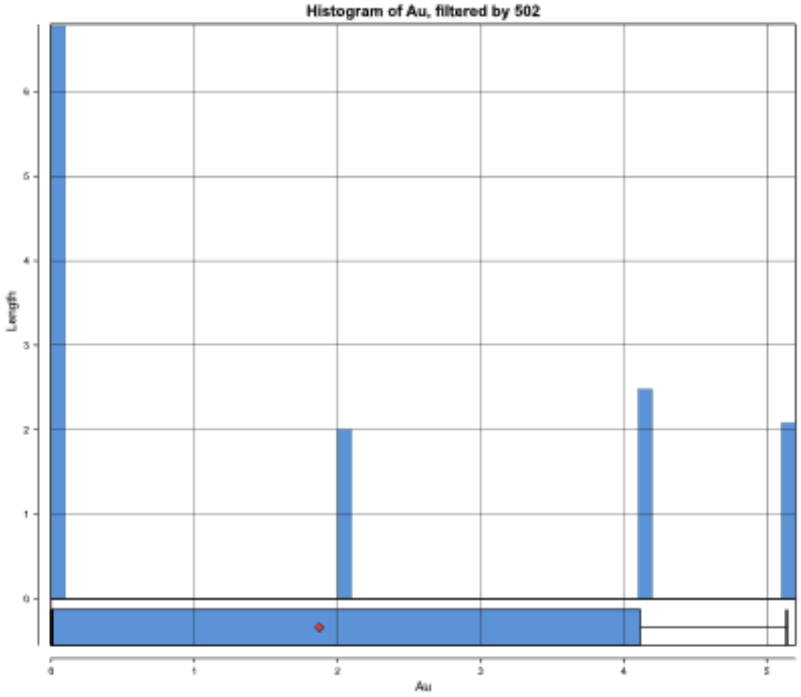
Middle Zone FW1 – 501

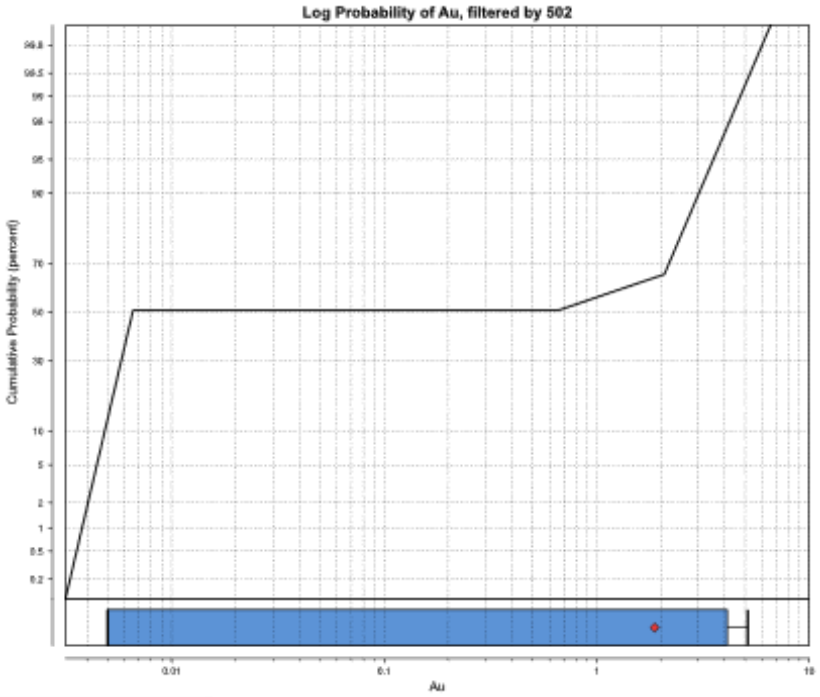




	Weighted Value
Count	8
Length	10.0
Mean	3.012
SD	3.876
CV	1.287
Variance	15.021
Minimum	0.005
Q1	0.008
Q2	2.100
Q3	4.270
Maximum	10.272

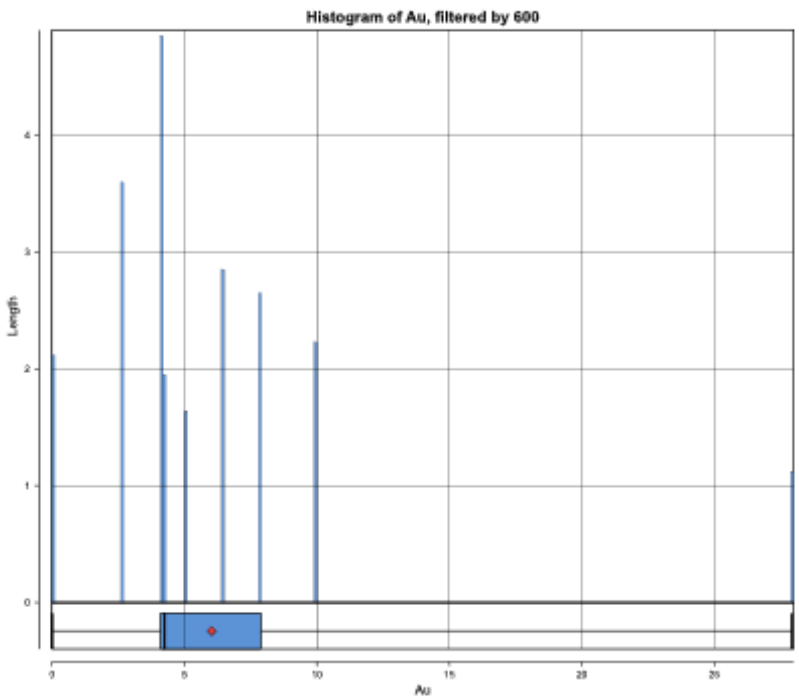
Middle Zone FW2 – 502

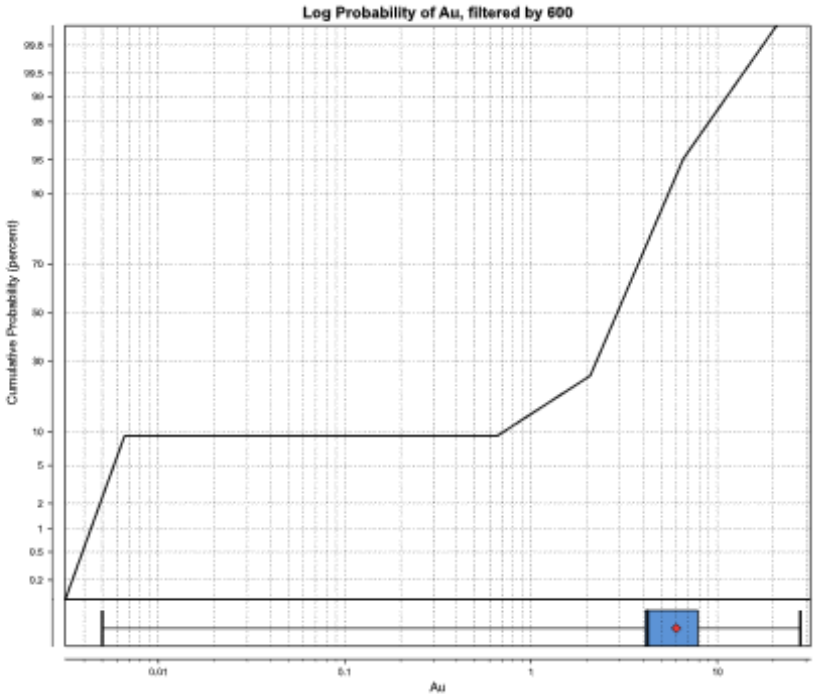




	Weighted Value
Count	6
Length	13.3
Mean	1.873
SD	2.294
CV	1.224
Variance	5.261
Minimum	0.005
Q1	0.005
Q2	0.005
Q3	4.117
Maximum	5.139

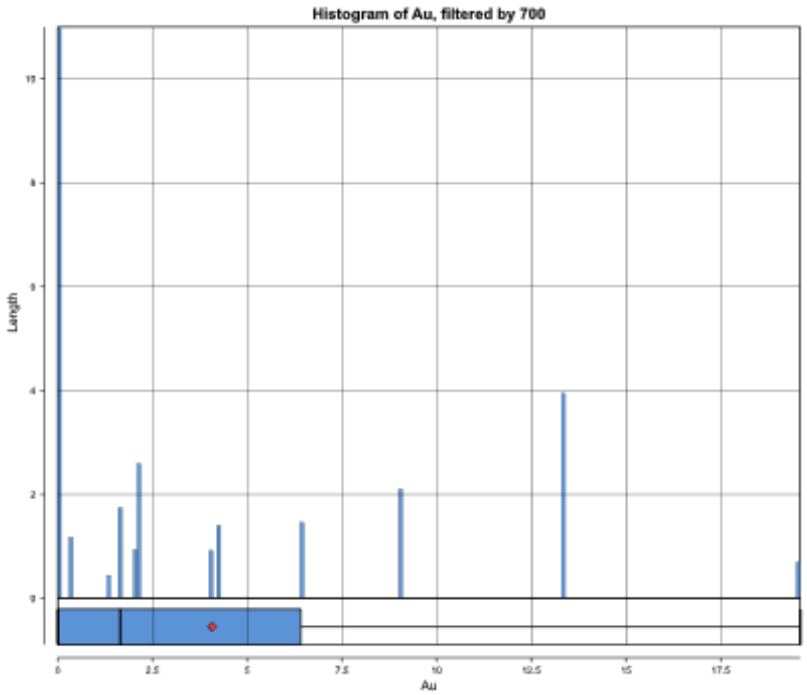
Middle Zone FW3 – 600

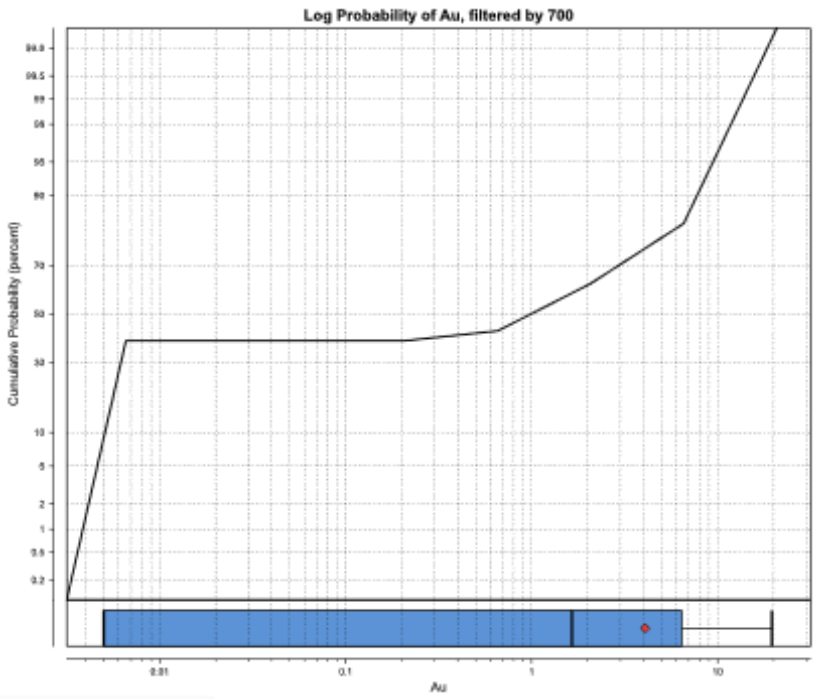




	Weighted Value
Count	10
Length	23.0
Mean	6.030
SD	5.945
CV	0.986
Variance	35.344
Minimum	0.005
Q1	4.104
Q2	4.259
Q3	7.889
Maximum	27.935

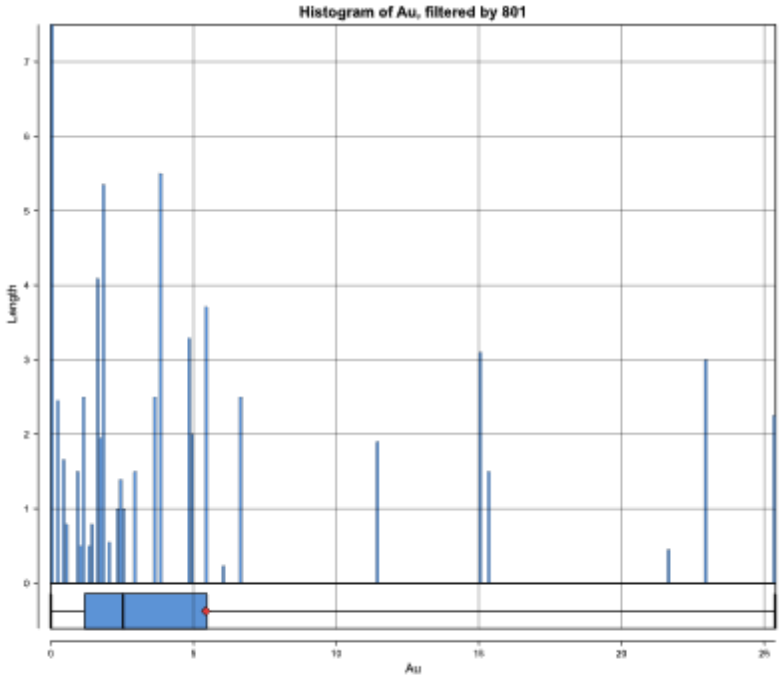
Middle Zone HW – 700

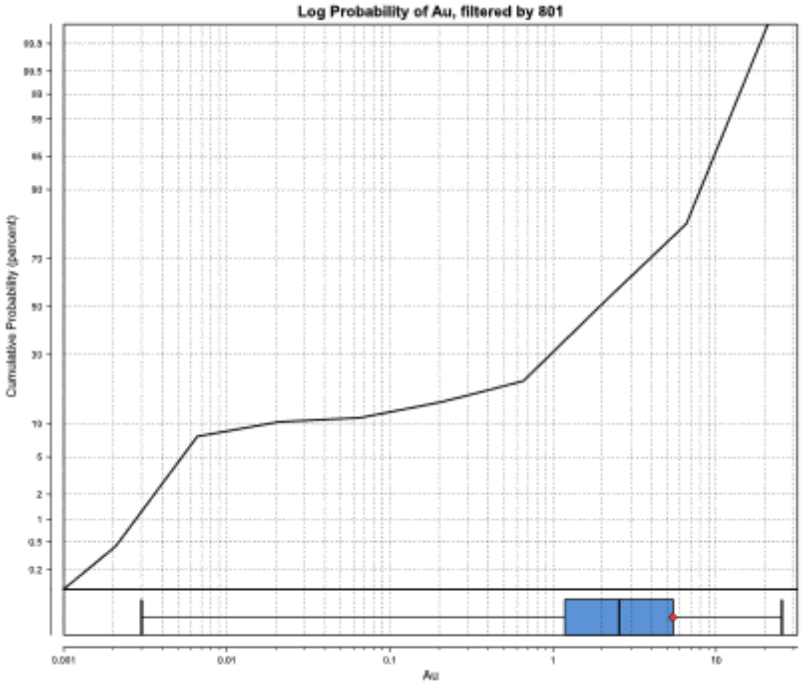




	Weighted Value
Count	18
Length	28.4
Mean	4.083
SD	5.473
CV	1.340
Variance	29.951
Minimum	0.005
Q1	0.005
Q2	1.653
Q3	6.423
Maximum	19.600

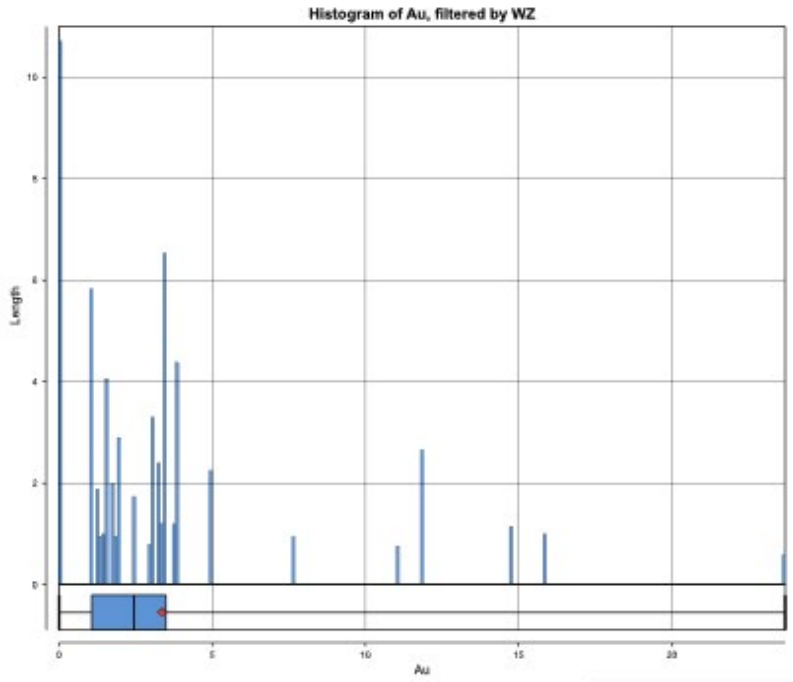
Wolf Zone – 801

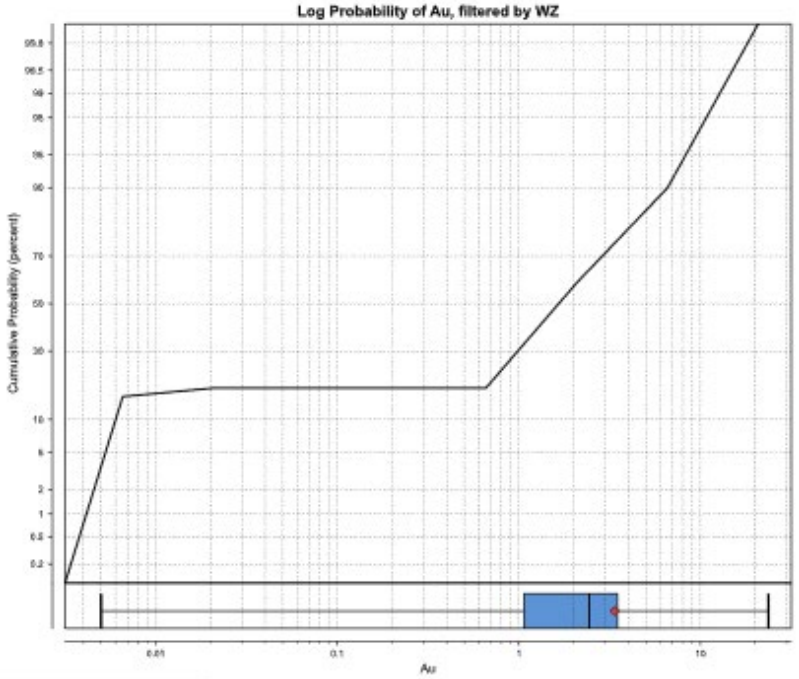




	Weighted Value
Count	48
Length	67.0
Mean	5.438
SD	6.942
CV	1.276
Variance	48.188
Minimum	0.003
Q1	1.180
Q2	2.540
Q3	5.477
Maximum	25.382

Wolf Zone – WZ



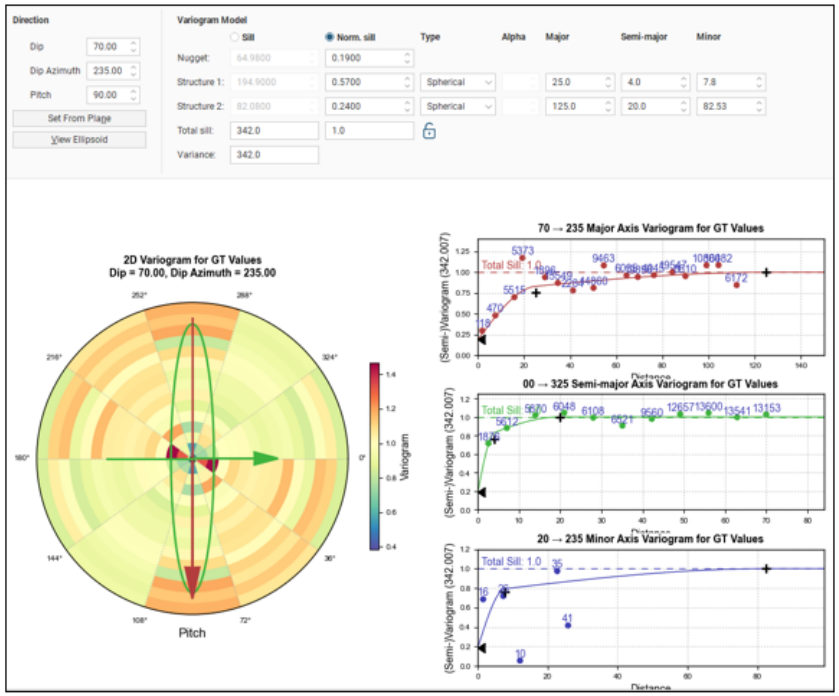


	Weighted Value
Count	39
Length	61.2
Mean	3.367
SD	4.125
CV	1.225
Variance	17.018
Minimum	0.005
Q1	1.065
Q2	2.442
Q3	3.495
Maximum	23.700

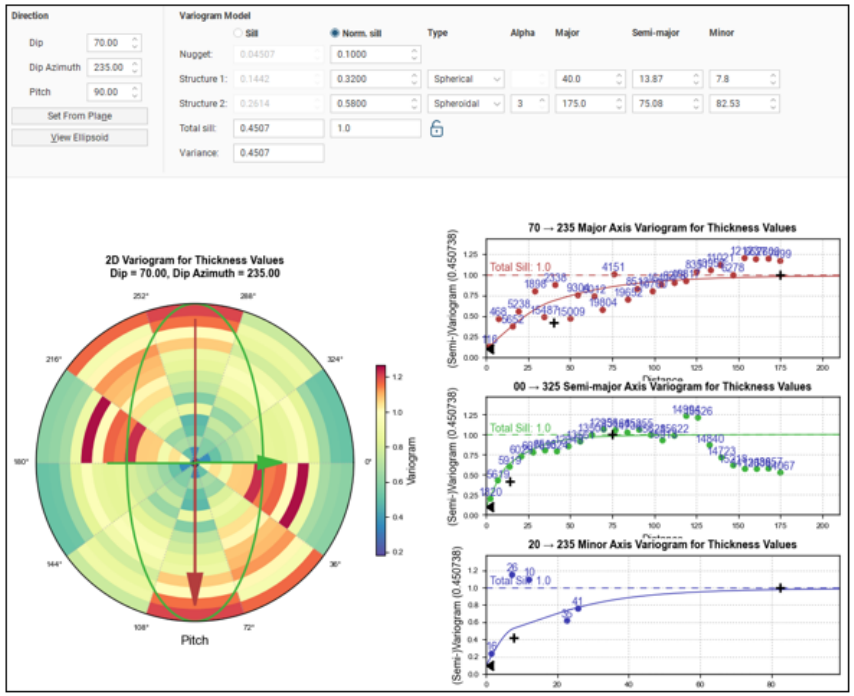
APPENDIX D

Variograms

LZ – Accumulation Variogram



LZ – Thickness Variogram



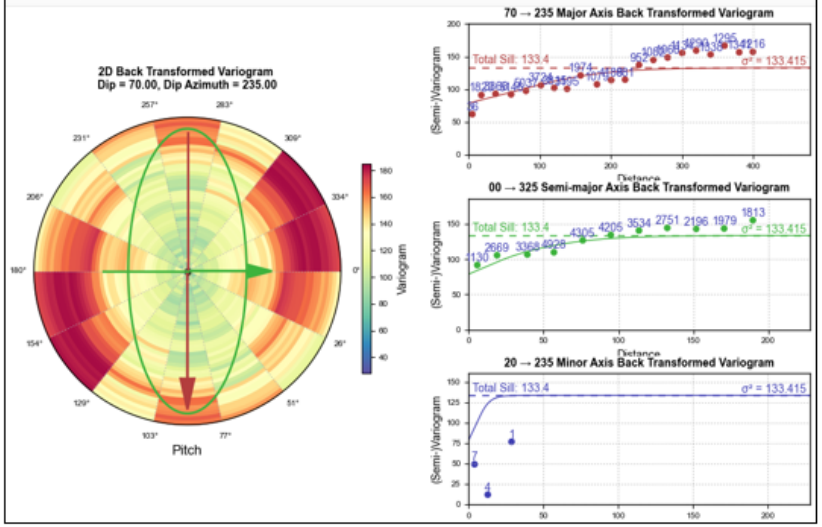
UZ – Accumulation Variogram

Back Transformed Variogram Model

	Sill	Normalised Sill	Type	Alpha	Major	Semi-major	Minor
Nugget:	78.7814	0.5905					
Structure 1:	16.6502	0.1248	Spheroidal	9	230.0	50.0	20.0
Structure 2:	37.9298	0.2843	Spheroidal	9	300.0	125.0	20.0
Total sill:	133.3614	0.9996					
Variance:	133.41468						

Direction

Dip: 70.0 Dip Azimuth: 235.0 Pitch: 90.0



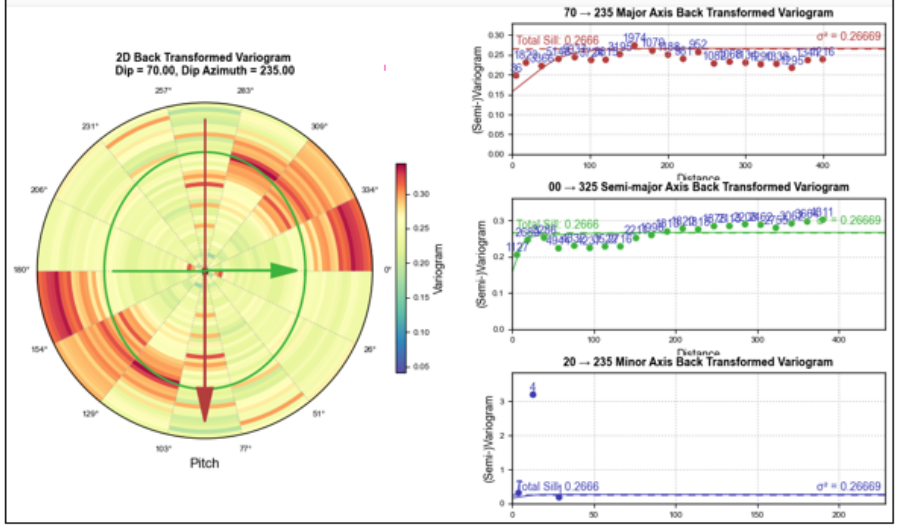
UZ – Thickness Variogram

Back Transformed Variogram Model

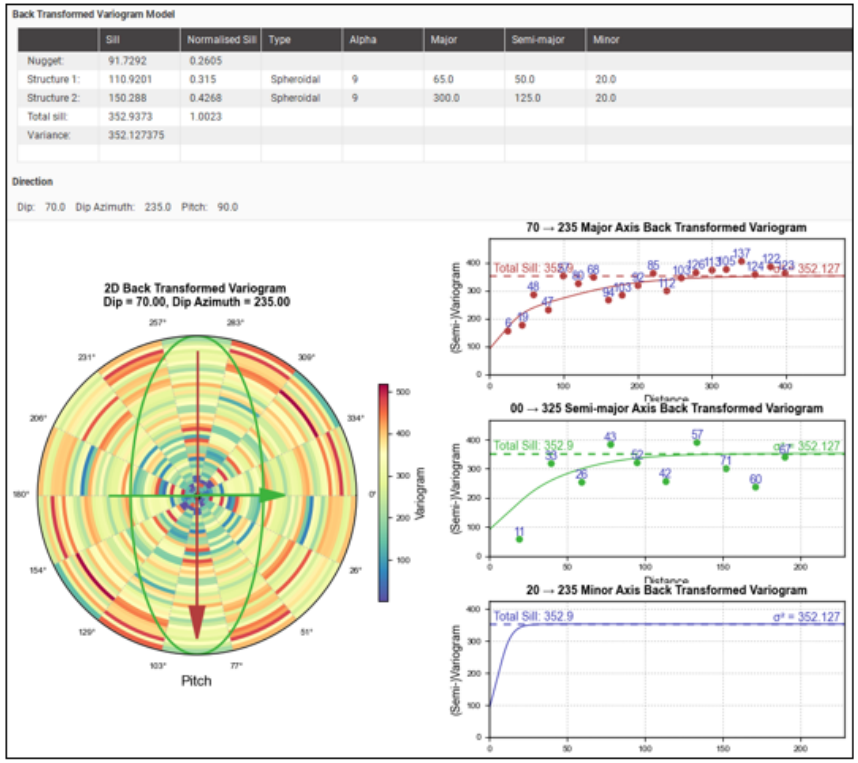
	Sill	Normalised Sill	Type	Alpha	Major	Semi-major	Minor
Nugget:	0.1575	0.5905					
Structure 1:	0.0976	0.3658	Spheroidal	9	95.0	22.0	20.0
Structure 2:	0.0115	0.0433	Spheroidal	9	200.0	170.0	20.0
Total sill:	0.2666	0.9996					
Variance:	0.26669						

Direction

Dip: 70.0 Dip Azimuth: 235.0 Pitch: 90.0



MZ – Accumulation Variogram



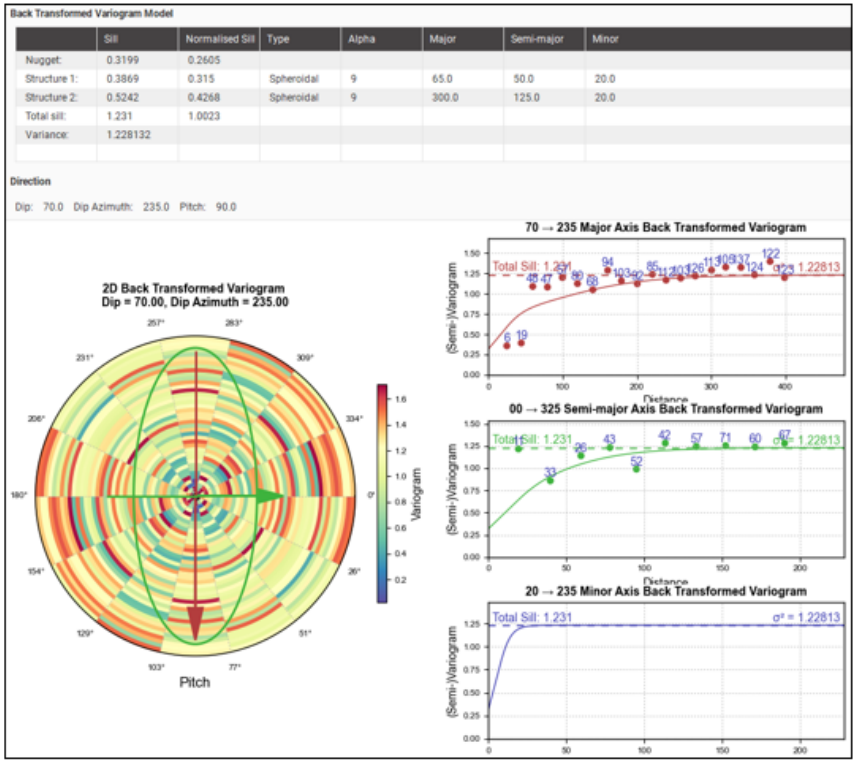
Back Transformed Variogram Model

	Sill	Normalised Sill	Type	Alpha	Major	Semi-major	Minor
Nugget:	91.7292	0.2605					
Structure 1:	110.9201	0.315	Spheroidal	9	65.0	50.0	20.0
Structure 2:	150.288	0.4268	Spheroidal	9	300.0	125.0	20.0
Total sill:	352.9373	1.0023					
Variance:	352.127375						

Direction

Dip: 70.0 Dip Azimuth: 235.0 Pitch: 90.0

MZ – Thickness Variogram



Back Transformed Variogram Model

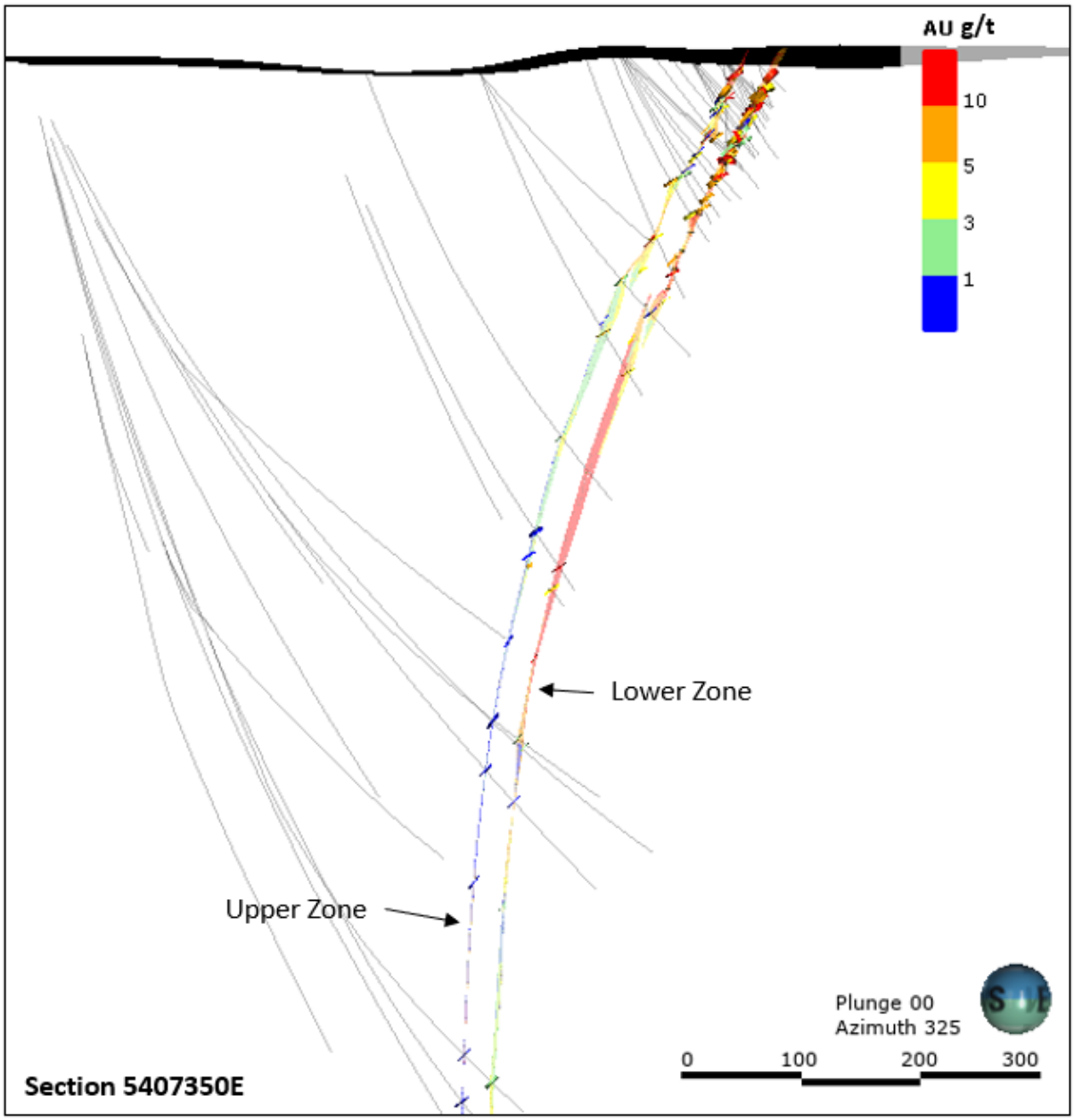
	Sill	Normalised Sill	Type	Alpha	Major	Semi-major	Minor
Nugget:	0.3199	0.2605					
Structure 1:	0.3869	0.315	Spheroidal	9	65.0	50.0	20.0
Structure 2:	0.5242	0.4268	Spheroidal	9	300.0	125.0	20.0
Total sill:	1.231	1.0023					
Variance:	1.228132						

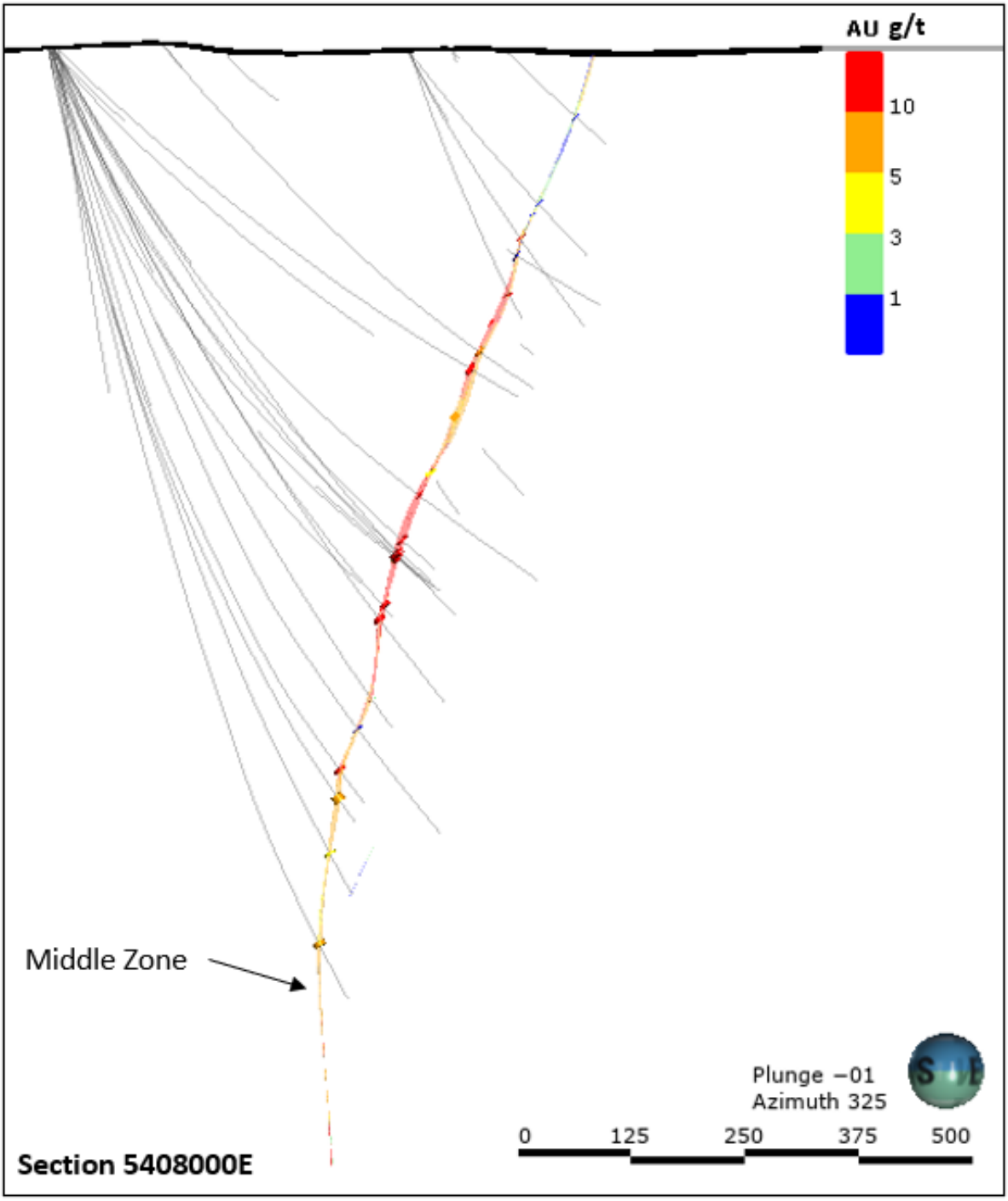
Direction

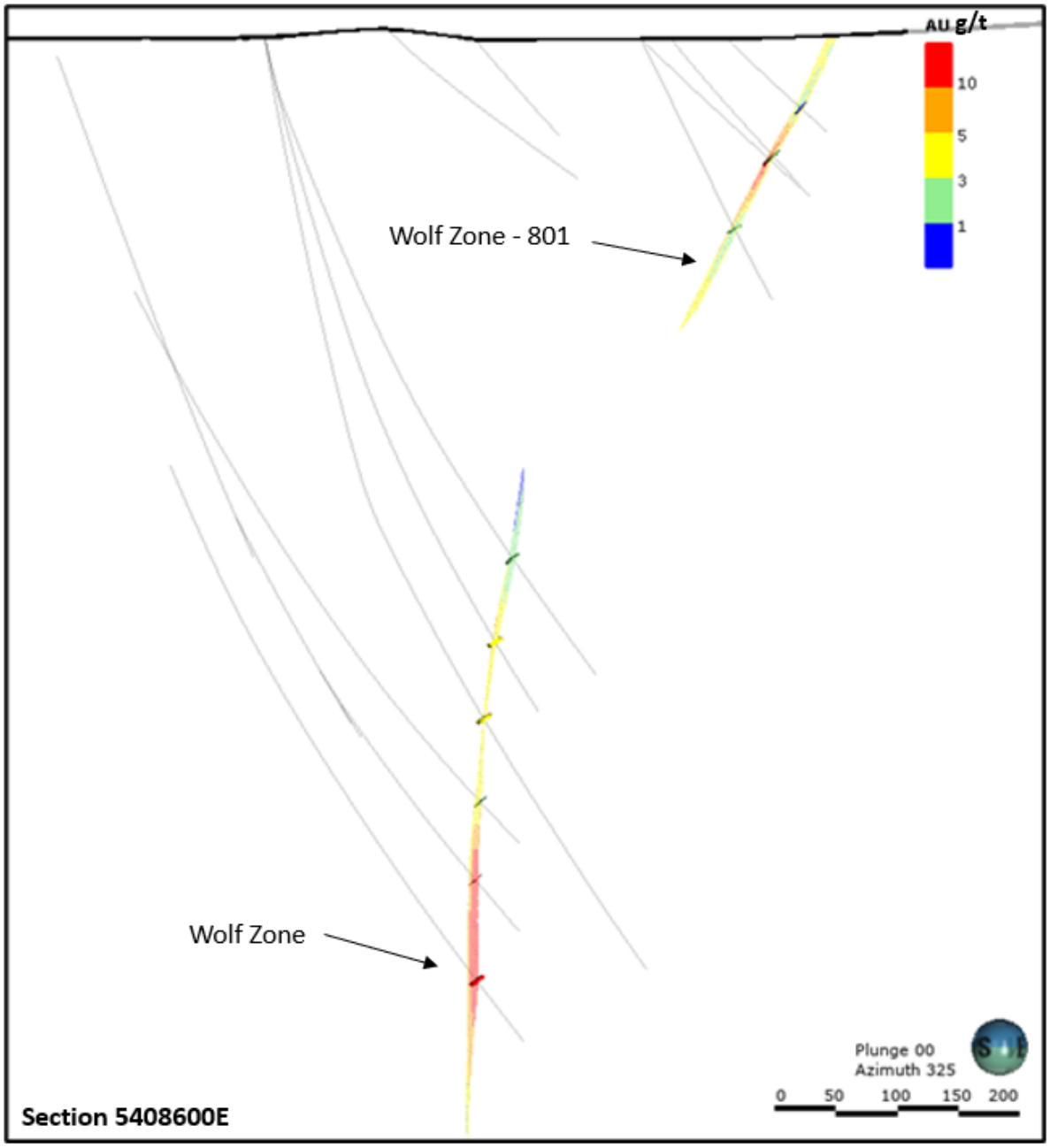
Dip: 70.0 Dip Azimuth: 235.0 Pitch: 90.0

APPENDIX E

Au Block Model Cross Sections

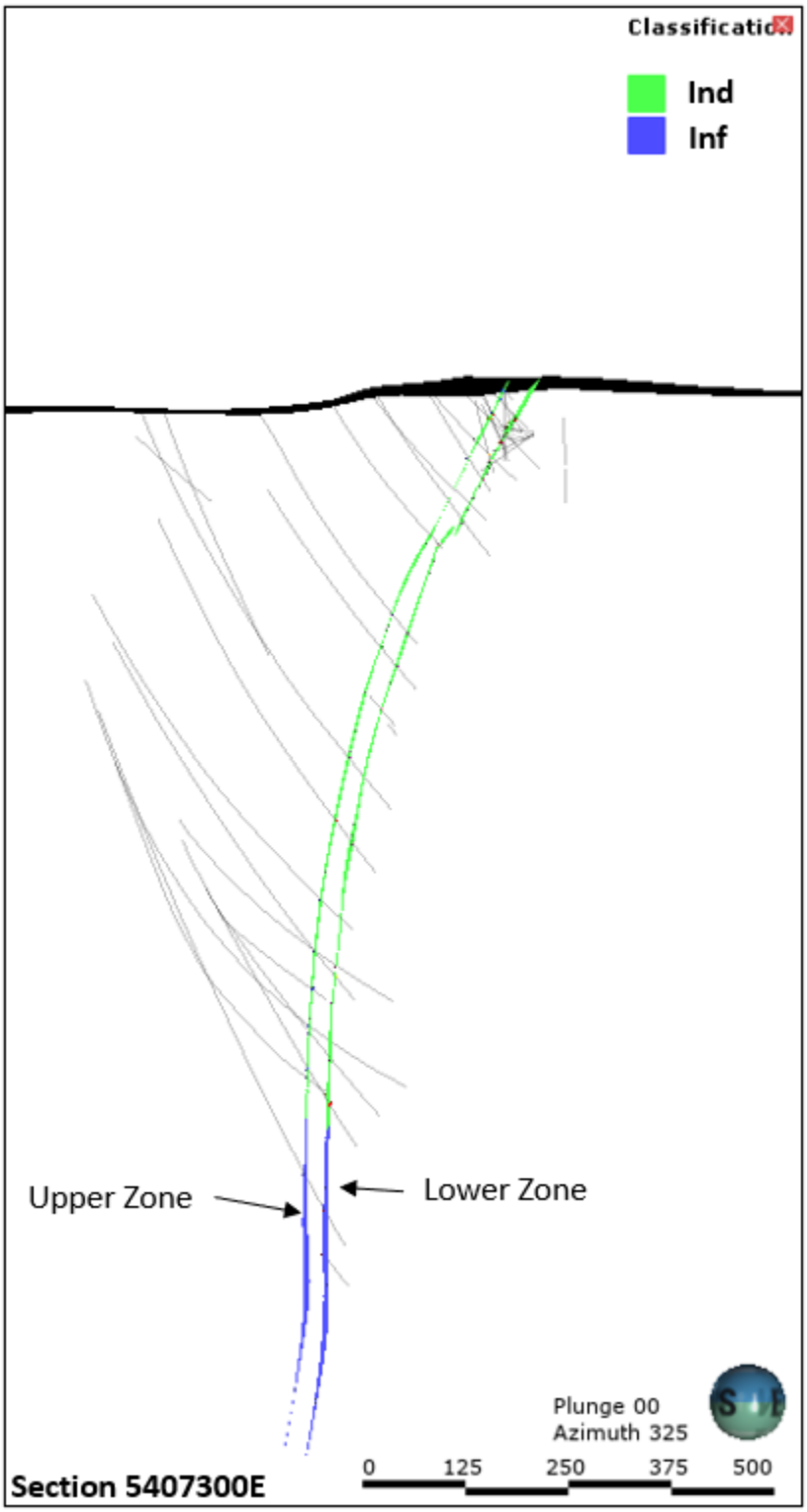

AU g/t
10
5
3
1
Lower Zone
Upper Zone
Plunge 00
Azimuth 325
0
100
200
300
Section 5407350E

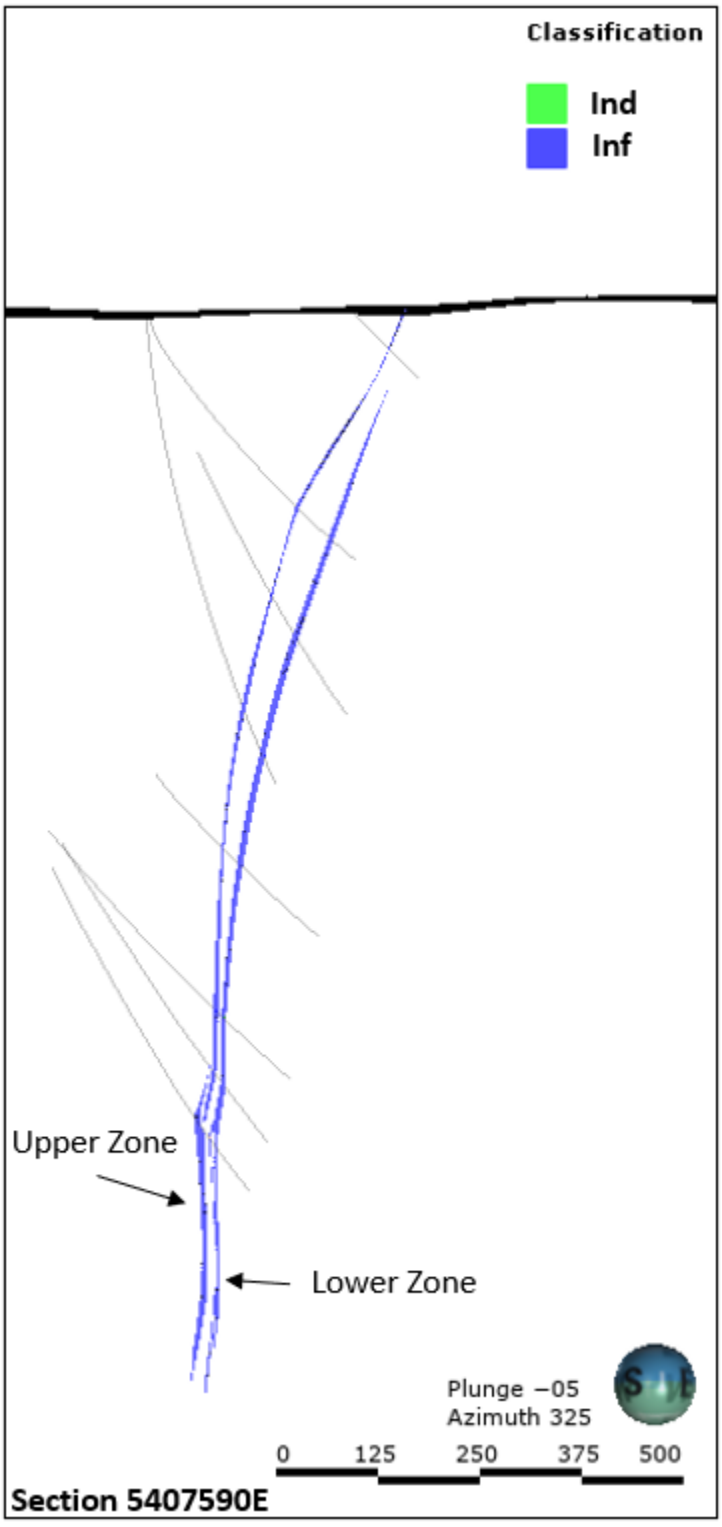

AU g/t
10
5
3
1
Middle Zone
Plunge −01
Azimuth 325
0
125
250
375
500
Section 5408000E

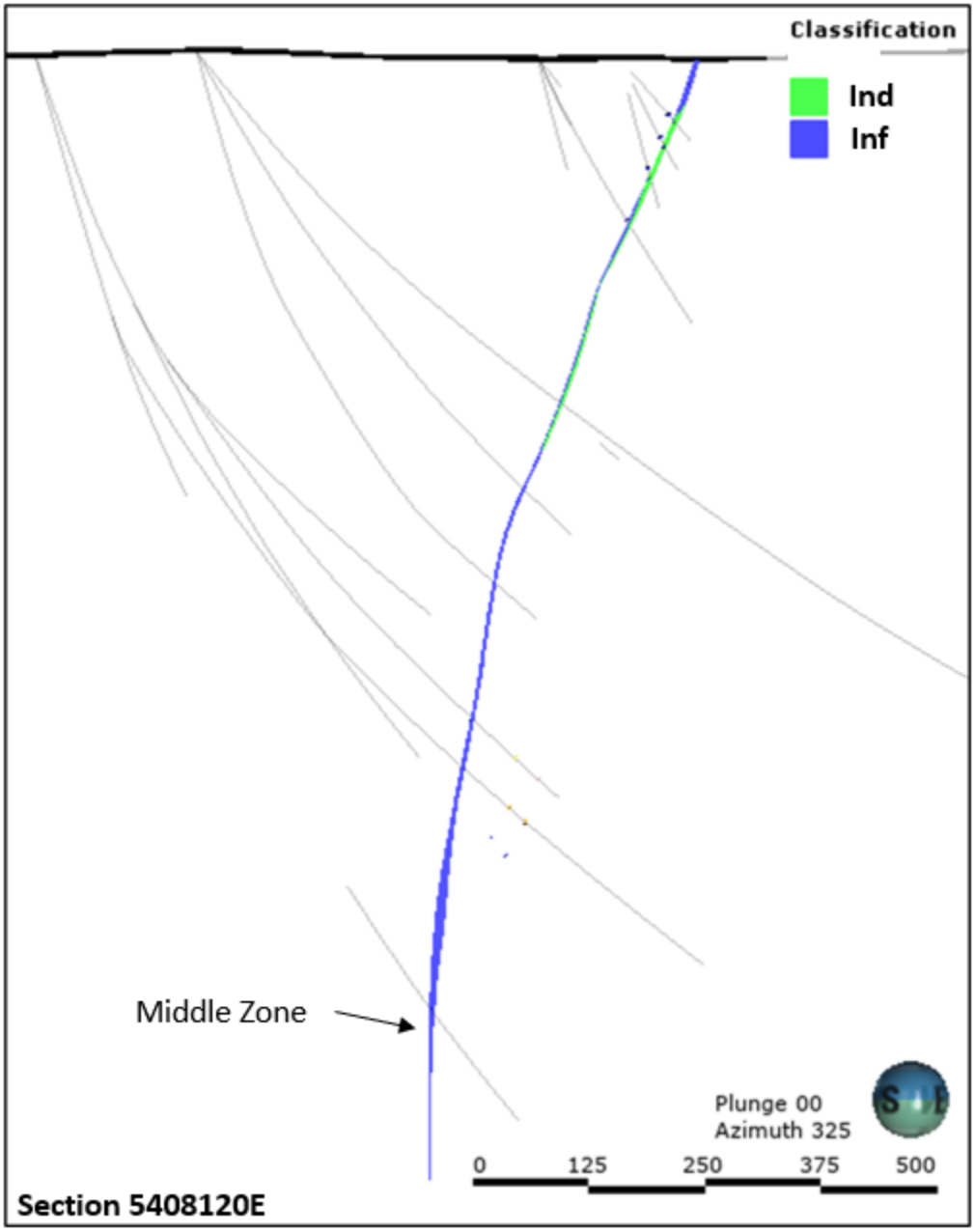

AU g/t
10
5
3
1
Wolf Zone - 801
Wolf Zone
Plunge 00
Azimuth 325
0 50 100 150 200
Section 5408600E

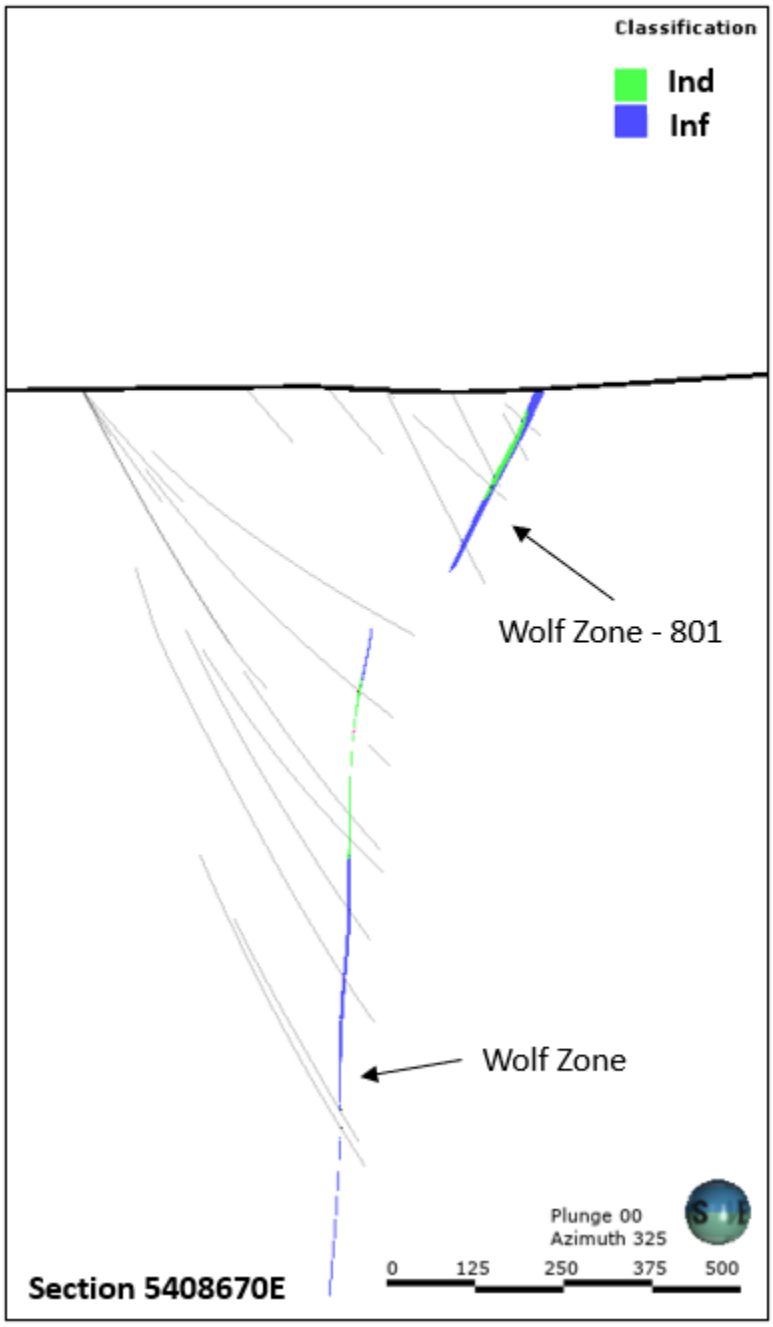
APPENDIX F

Classification Block Model Cross Sections


Classificatio
Ind
Inf
Upper Zone
Lower Zone
Plunge 00
Azimuth 325
0
125
250
375
500
Section 5407300E


Classification
Ind
Inf
Upper Zone
Lower Zone
Plunge −05
Azimuth 325
0
125
250
375
500
Section 5407590E


Classification
Ind
Inf
Middle Zone
Plunge 00
Azimuth 325
0
125
250
375
500
Section 5408120E


Classification
Ind
Inf
Wolf Zone - 801
Wolf Zone
Plunge 00
Azimuth 325
0
125
250
375
500
Section 5408670E