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NI 43-101 Technical Report, Preliminary Economic Assessment of the Crean Hill Project, Sudbury, Ontario, Canada

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This notice is an integral component of the NI 43-101 Technical Report, Preliminary Economic Assessment of the Crean Hill Project, Sudbury, Ontario, Canada (Technical Report or Report) and should be read in its entirety and must accompany every copy made of the Technical Report. The Technical Report has been prepared in accordance with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects.

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DATE AND SIGNATURE PAGE

Report title: NI 43-101 Technical Report, Preliminary Economic Assessment of the Crean Hill Project, Sudbury, Ontario, Canada

Property: Crean Hill Project

Location: Sudbury, Ontario, Canada

Effective Date of Report: July 31, 2023

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A3477	Sodium Diisobutyl Dithiophosphate
AACE	The Association for the Advancement of Cost Engineering
ABA	Acid Base Accounting
ADEX	Advanced Exploration
Ai	Abrasion Index
As	Arsenic
Au	Gold
BHEM	Borehole Electromagnetics
BWi	Ball Mill Work Index
CAD	Canadian Dollar
CCA	Capital Cost Allowance
CDE	Canadian Development Expense
Cu	Copper
CORe	Challenging Ore Recovery
COG	Cut Off Grade
CoV	Cut Off Value
CRF	Cemented Rockfill

CRM	Certified Reference Material
EGRG+PGE	Extended Gravity Recoverable Gold and PGE
EM	Electromagnetic
EPCM	Engineering, Procurement, Construction Management
FoS	Factor of Safety
FCS	Full Circuit Simulation
FEL	Front End Loader
FW	Footwall
G&A	General and Administrative
HW	Highwall
ICP	Induced Couple Plasma Mass Spectroscopy
ICP-AES	Inductively Coupled Atomic Emission Spectrometry
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ID2	Inverse Distance Squared
INCO	International Nickel Company
IP	Induced Polarization
IQD	Inclusion-Rich Quartz Diorite
IRR	Internal Rate of Return
ktpa	Kilo Tonnes Per Annum
LCT	Lock Cycle Test
LHD	Load Haul Dumper
LIMS	Laboratory Information Management System
LSHPM	Low Sulphide, High Precious Metal
LTE	Long Term Evolution
Magna	Magna Mining Inc.
masl	Meters Above Sea Level
MEBS	Mines Exploration Borehole System
MENDM	Ministry of Energy, Northern Development and Mines
MIMDAS	MIM Distributed Acquisition System
MRE	Mineral Resource Estimate
Mt	Million metric tonnes
MT	Million US tons
MTBS	Metabasalt
MTpa	Million US tons per annum
Mtpa	Million metric tonnes per annum
NAG	Net Acid Generation
Ni	Nickel
NI 43-101	National Instrument 43-101
NPV	Net Present Value
NSR	Net Smelter Return
NTS	National Topographic System
OSA	Overall Slope Angle
Pd	Palladium
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PGE	Platinum-Group Elements
PGM	Platinum Group Metals
PMD	Post Mining Designation
ppm	Parts Per Million

Pt	Platinum
QD	Quartz Diorite
QP	Qualified Person(s)
QTO	Quantity Take Offs
RF	Unconsolidated Rockfill
RQD	Rock Quality Designation
RWi	Rod Mill Work Index
SAG	Semi-Autogenous
SCC	Standards Council of Canada
SG	Specific Gravity
SIBX	Sodium Isobutyl Xanthate
SIC	Sudbury Igneous Complex
TPM	Total Precious Metals
UG	Underground
UTM	Universal Transverse Mercator
Vale	Vale Canada Limited
VCR	Vertical Crater Retreat
VLF	Very Low Frequency
VRM	Vertical Retreat Mining
WRD	Waste Rock Dump
XRT	X-Ray Transmission

1 SUMMARY

1.1 INTRODUCTION

Magna Mining Inc. (Magna) is a Canadian exploration and development company, based in Sudbury, Canada and is publicly listed on the TSX Venture Exchange (TSXV: NICU). Magna is Sudbury-focused with a vision of bringing new as well as past producing mines into production, including the development of Magna's 100% owned Crean Hill Ni-Cu-platinum group metals project (Crean Hill Project or the Project) near Sudbury, Canada.

Magna retained independent consultants to prepare a technical report for a Preliminary Economic Assessment (PEA) on the Crean Hill Project. The technical report provides an update of the mineral resources and preliminary assessment of the economics for the Project. The reporting of mineral resources complies with all disclosure requirements set out in the NI 43-101 Standards of Disclosure for Mineral Projects.

The following independent consultants were the primary contributors to this PEA:

- Stantec Consulting Ltd. (Stantec)
- SGS Canada Inc. (SGS)
- XPS – Expert Process Solutions

1.2 PROPERTY LOCATION AND DESCRIPTION

The Project is in Denison Township within the City of Greater Sudbury, Ontario, Canada, approximately 30 kilometres southwest of downtown Sudbury. The Project is centered at approximately 46° 25.8' N latitude, 81° 21.1' W longitude (or 473,000 m E; 5,141,800 m N in NAD83 UTM [Universal Transverse Mercator] Zone 17N). The Project location is shown on a map of Ontario in Figure 1-1.

Figure 1-1: Crean Hill Project Location

The Project is accessible year-round by road from the nearby Trans Canada Highway 17, Regional Road 4, and an all-season gravel road. The location of the project relative to Sudbury and Highway 17 is shown in Figure 1-2.

Figure 1-2: Project Location Relative to Sudbury and Highway 17

The Project is a historical mine site that operated from the early 1900's until 2002. After operations ceased in 2002, the site was decommissioned and buildings, structures, and infrastructure were removed. The historic open pits have been backfilled and openings to the surface such as shafts and ventilation raises have been capped. A portion of the site has been used to stockpile waste rock from other nearby operations. The main site roads continue to be maintained to allow controlled access to environmental sampling points and to access roads to other sites in the area. An aerial view of the current site is shown in Figure 1-3.

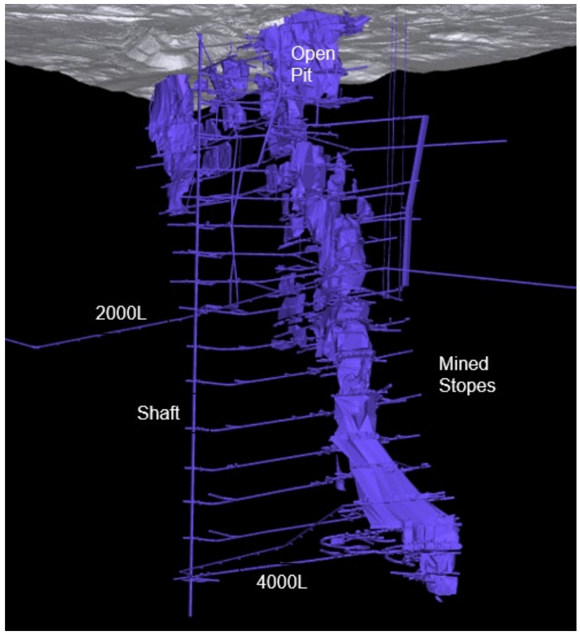
Figure 1-3: Aerial View of Current Project Site (not to scale)

1.3 HISTORY

The Crean Hill deposit was first discovered in 1885 and production was first reported in 1906. From 1906 to 2002 the site was subject to sporadic production and exploration by various operators. During this period, approximately 20 million tonnes of ore were produced at an average grade of 1.31% Ni and 1.09% Cu.

The existing Crean Hill underground mine workings have been flooded since operations ceased in 2002. The original shaft was approximately 2,000 ft deep to just below the 2000 Level (2000L) and was later deepened to just below the 4000L (approximately 4,180 ft below surface). The shaft is understood to have been equipped with five compartments (two skips, a cage, a counterweight, and services/ladderway). The historic sublevels in the upper mine above 2000L were spaced at approximately 200 ft intervals and developed from the shaft as track drifts. There was no internal ramp system connecting the levels. In the upper levels, the shrinkage stoping mining method was used, and stopes are assumed by the QP to be left open or partially backfilled with unconsolidated waste rock. Below 2000L, the primary mining method was Vertical Retreat Mining (VRM) also known as Vertical Crater Retreat (VCR), with sublevel spacing ranging from approximately 250 to 300 ft and no internal ramp system. The VRM stopes were mined as primaries and secondaries with hydraulic sandfill used to fill the voids. The 4000L area included a crusher and conveyor to the shaft loading pocket. The existing lateral development excavations outside the historic stopes are assumed by the QP to be accessible, however all previous underground infrastructure is assumed by the QP to be not available. The historic existing Open Pit and Underground mine workings are shown in Figure 1-4.

Figure 1-4: Existing Open Pit and Underground Mine Workings (isometric view)



1.4 GEOLOGICAL SETTING AND MINERALIZATION

Ni-Cu-Platinum Group Elements (PGE) deposits in Sudbury occur within the Sudbury Structure that formed as a result of a major early Proterozoic meteorite impact 1,850 million years ago (Ames and Farrow, 2007). The Sudbury Structure straddles the unconformity between Archean gneisses and plutons of the Superior Province and overlying Paleoproterozoic Huronian supra-crustal rocks of the Southern Province. It is geographically divided into the North, South, and East Ranges and comprises four geologic domains. The Property is in the South Range of the Sudbury Igneous Complex (SIC). The main mass of the South Range SIC consists of a lower unit of the quartz-rich norite. Stratigraphically above is the Green Norite with irregular bodies of Brown Norite followed by the Quartz Gabbro then the Granophrye layers. There is a magmatic breccia called Sublayer found at the basal contact of the main mass in embayment and trough structures. The footwall (FW) to the SIC South Range is the Southern Province. The geology can roughly be divided into the Early Proterozoic (~2,450 Ma) Murray and Creighton Granite Plutons and Huronian Supergroup (2,250 to 2,460 Ma) mafic and felsic volcanic and sedimentary rocks. The Creighton and Murray Plutons are intrusive into older Huronian volcanic and sedimentary rocks, mostly of the Elsie Mountain and Stobie Formations.

The South Range of the SIC and adjacent Huronian rocks, for the most part, dip vertically or steeply north or south. Stratigraphic tops generally face south, away from the SIC and toward the Grenville Front. The South Range Shear zone and Creighton and Murray faults are the result of the deformation events that have shaped the present-day South Range. The age of the deformation which has resulted in the current sub-vertical orientation of the Huronian rocks has not been definitively established. The metasedimentary rocks are interbedded sparingly with mafic volcanic flows of the Elsie Mountain Formation and commonly with volcanic rocks of the Stobie Formation. Many of these interflow metasedimentary rocks are sulphide-bearing. The sulphides are dominantly pyrrhotite with minor amounts of pyrite and trace chalcopyrite. South Range footwall rocks are cut by several small diabase and gabbroic intrusions that are often difficult to distinguish in the field. These include Matachewan dykes, Nipissing intrusions, quartz diabase (trap dykes), and Olivine Diabase. Both the quartz diabase and olivine diabase dykes are younger than the SIC. The Archean and early Proterozoic basement rocks are all crosscut by Sudbury Breccia. There are several main types of mineral deposits in the Sudbury area:

- Contact deposits, including massive sulphide consisting of nickel, copper, cobalt, platinum, palladium, and gold mineralization along the lower contact of the SIC, both within the contact sublayer and in the immediately adjacent Footwall Breccia.
- Footwall deposits, including sulphide veins and stringers containing copper, nickel, platinum, palladium, and gold in the brecciated footwall rocks beneath the SIC.
- Structurally and/or hydrothermally remobilized sulphide nickel, copper, cobalt, platinum, palladium, and gold mineralization.
- Offset dyke deposits, including massive sulphide consisting of nickel, copper, cobalt, platinum, palladium, and gold mineralization associated with brecciated and inclusion bearing phases (IQD) of the quartz diorite offset dykes (QD).
- Hybrid type deposits representing combinations of the above.

The Property straddles the South Range of the SIC approximately 30 km southwest of Sudbury, in Denison Township. From 1906-2002 a total of 20,370,000 tonnes of ore grading 1.09% Cu, 1.31% Ni, 1.56 g/t TPM (total precious metals) was produced from the Main, Intermediate and 9400 zones. The Property hosts part of a large trough structure at the base of SIC which contains several previously mined ore deposits including Crean Hill Main Orebody, Crean Hill Intermediate Orebody, Crean Hill West (9400) Orebody, Ellen Mine, and Lockerby Mine, each sitting in embayments within the larger trough. The embayments largely control the distribution of Ni-Cu mineralization. Much of the historic mining activity on the property exploited Contact Type deposits. Mineralization includes blebby to massive accumulations of sulphide, including pyrrhotite > chalcopyrite > pentlandite concentrated within embayment depressions along the base of the SIC, both within the contact sublayer and in the immediately adjacent FW Breccia.

A significant portion of the mineralization, such as the 109 FW (footwall) Zone, the 101 FW Zone, and part of the 9400 Zone, are hosted in the footwall rocks. The host rocks are dominated by metamorphosed basalt (historically mapped and logged as greenschist), but also include gabbro, andesite, rhyolite, and sedimentary units (arkosic quartzite and meta-pelite) of the Huronian supergroup, Elsie Mountain Formation. Minor lithologies include olivine diabase, quartz diabase (trap dykes), granite, schist, amphibolite, and Sudbury Breccia in the footwall, and quartzose norite at the SIC contact. The main mineralized zones from east to west are as follows:

- 109 W/Remnant Zones
- 126
- 123
- 109 FW
- 109 HW
- 99 Zone
- 101
- 9400

- 9400 FW Extension

1.5 EXPLORATION AND DRILLING

As of the effective date of the current Mineral Resource Estimate a total of 4,009 drillholes totalling 515,664 m (1,691,812 ft) make up the Crean Hill drillhole dataset, prior to Magna acquiring the property in late 2022. Magna has completed 68 surface diamond drill holes between Q4 2022 and Q3 2023 for a total of 12,180 m. Drilling to date has been designed to enhance Magna's understanding of the near surface Intermediate, 101 FW, and 109 FW zones both along strike and down-dip of historical mining areas. Results of the Magna drilling have not yet been considered in the deposit interpretation, and assay results have not been incorporated into the mineral resource estimate.

1.6 MINERAL RESOURCES

Completion of the updated mineral resource estimates (MREs) for the Property involved the assessment of a drill hole database, which included all data for surface drilling completed through the end of 2017, as well as three-dimensional (3D) mineral resource models (resource domains), 3D models of all mined-out areas (open pit and underground), 3D models of cross-cutting dykes, a recent topographic surface, and available written reports.

The Inverse Distance Squared (ID2) calculation method restricted to mineralized domains was used to interpolate grades for Ni (%), Cu (%), Co (%), Pt (g/t), Pd (g/t), and Au (g/t) into block models.

Indicated and inferred mineral resources are reported in the summary tables presented below. The current MRE takes into consideration that the Projects deposits may be mined by open pit and underground mining methods. The MRE for the Property are prepared and disclosed in compliance with all current disclosure requirements for mineral resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). The classification of the current MRE's into Inferred is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves, including the critical requirement that all mineral resources "have reasonable prospects for eventual economic extraction". The current MRE for the Deposit is presented in Table 1-1 and includes an in-pit and an underground (below-pit) Mineral Resource (estimated from the bottom of the 2022 pit).

Table 1-1: Crean Hill Deposit In-Pit (A) and Underground (below-pit) (B) Mineral Resource Estimate, August 19, 2022

Cut-off Grade	Tonnes	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	NiEq %
In-Pit Mineral Resource Estimate								
Indicated								
0.3% NiEq	16,760,000	0.53	0.49	0.02	0.48	0.37	0.25	1.08
Inferred								
0.3% NiEq	434,000	0.43	0.49	0.02	0.29	0.14	0.07	0.82
Underground (Below-Pit) Mineral Resource Estimate								
Indicated								

Cut-off Grade	Tonnes	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	NiEq %
1.1% NiEq	14,531,000	0.96	0.84	0.03	0.88	1.02	0.54	2.07
Inferred								
1.1% NiEq	1,170,000	0.61	0.46	0.02	0.64	1.09	0.21	1.41

1. The classification of the current Mineral Resource Estimate into Indicated and Inferred is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.
2. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
3. All Resources are presented undiluted and in situ, constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction.
4. Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
5. It is envisioned that parts of the Denison deposit may be mined using open pit mining methods. In-pit mineral resources are reported at a cut-off grade of 0.3 % NiEq within a conceptual pit shell.
6. The results from the pit optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the Property. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.
7. Underground (below-pit) Mineral Resources are estimated from the bottom of the pit and are reported at a base case cut-off grade of 1.1 % NiEq. The underground Mineral Resource grade blocks were quantified above the base case cut-off grade, below the constraining pit shell and within the constraining mineralized wireframes. At this base case cut-off grade the deposit shows good deposit continuity with limited orphaned blocks. Any orphaned blocks are connected within the models by lower grade blocks.
8. Based on the size, shape, location and orientation of the Denison deposit, it is envisioned that the deposit may be mined using longhole open stoping (a bulk mining method that has long been utilized in the Sudbury region).
9. High grade capping was done on 10 ft (3.05 m) composite data.
10. Bulk density values were determined based on physical test work from each deposit model and waste model.
11. NiEq Cut-off grades are based on metal prices of $8.50/lb Ni, $3.752/lb Cu, $22.00/lb Co, $1000/oz Pt, $2000/oz Pd and $1,750/oz Au and metal recoveries of 78% for Ni, 95.5% for copper, 56% for Co, 69.2% for Pt, 68% for Pd and 67.7% for Au.
12. The in-pit base case cut-off grade of 0.3% NiEq considers a mining cost of US$2.50/t rock and processing, treatment and refining, transportation and G&A cost of US$38.00/t mineralized material, and an overall pit slope of 55 degrees. The below-pit base case cut-off grade of 1.1 % NiEq considers a mining cost of US$80.00/t rock and processing, treatment and refining, transportation and G&A cost of US$42.50/t mineralized material.
13. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading. The Author is not aware of any known mining, processing, metallurgical, environmental, infrastructure, economic, permitting, legal, title, taxation, socio-political, or marketing issues, or any other relevant factors not reported in this technical report, that could materially affect the current Mineral Resource Estimate.

1.7 MINERAL PROCESSING AND METALLURGICAL TESTING

The deposit is made up of the remnants of the historic mine operations of Crean Hill and extensions into the FW adjacent to the historic mining. The geometallurgical types typical in the Sudbury Basin are represented in the deposit. Approximately 30% of the mineralization consists of FW and LSHPM material identified in the 109 FW and 9400 FW adjoining the historic deposits.

The 109 FW material has been the subject of numerous evaluations, which indicated typical results, which are discussed in the report. The contact ore, which represents approximately 70% of the mineralization, is contained in extensions and remnants of the Ni mines and represents contact type material. Contact mineralization responds to flotation, and recovery of Ni is dependent on pyrrhotite content. The contact mineralization of the resource has not yet been tested and will be evaluated in subsequent phases of this project.

1.8 MINING

The mine design includes both an underground mine and open pit mine.

1.8.1 Underground Mine

The existing historic Crean Hill underground mine voids (including the open pits that are connected to underground workings) are either backfilled with waste rock or sandfill or flooded with water. The flooded workings will be dewatered ahead of advancing development by lowering a pump in the existing shaft. The water will be treated through the existing treatment system at site.

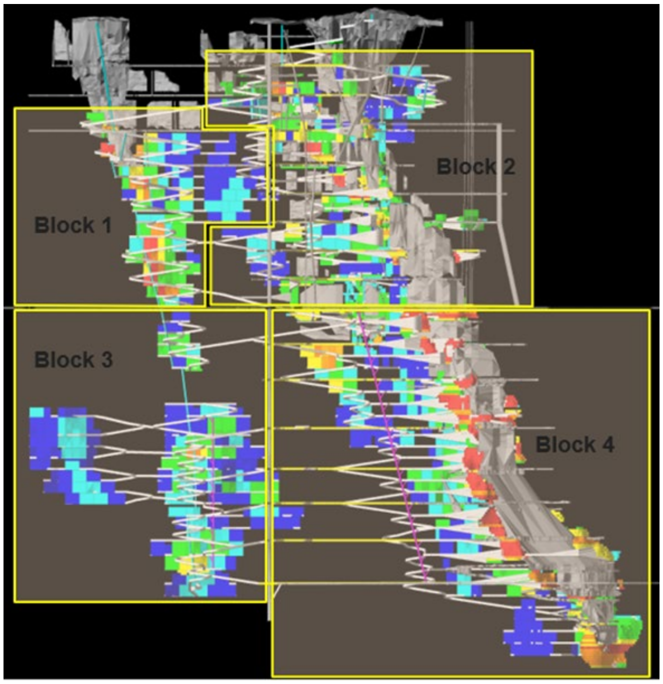
Access to the underground workings will be established in a phased approach. Initially, a boxcut and portal will be constructed, and an access ramp developed to support an Advanced Exploration (ADEX) program that includes an underground bulk sample and diamond drilling. Following the ADEX program, the access ramp will be extended to support initial production from the upper portion of the mine (above 2000L) using the ramp for material handling, while the existing shaft is being reconditioned and re-equipped for production and servicing the mine below 2000L down to 4000L. The purpose and use of the existing shaft will evolve as the Project and mine life progresses from an initial means for dewatering, to egress and ventilation while mining above 2000L, to a full production and service shaft for mining to below 4000L.

All ramp and lateral excavations will be developed using conventional drill and blast methods and diesel-powered mobile equipment. A mining contractor will complete the ADEX program and sufficient project capital development above 2000L to sustain 60% of Phase 1 steady state production before being replaced by owner personnel and equipment.

Both bulk and selective mining methods have been proposed. Longitudinal Longhole Stoping (Longhole) will be the primary mining method used in greenfield areas, and variations of Mechanized Cut and Fill (MCAF) will be used in proximity to the historic stopes. The stopes will be backfilled with a combination of cemented rockfill and unconsolidated rockfill. For Longhole mining, a Net Smelter Return (NSR) value of $123.50 was used as a stope cut-off value targeted in the mine plan, with a $100.00 NSR cut-off used for identifying incremental longhole stopes. For MCAF mining, a $134.50 NSR cut-off was used as a stope cut-off value targeted in the mine plan with $112.50 used for identifying incremental stopes.

The underground mine has been divided into four main blocks. The blocks are defined laterally by the shaft and vertically by the 2000L as shown in Figure 1-5.

Figure 1-5: Mining Blocks



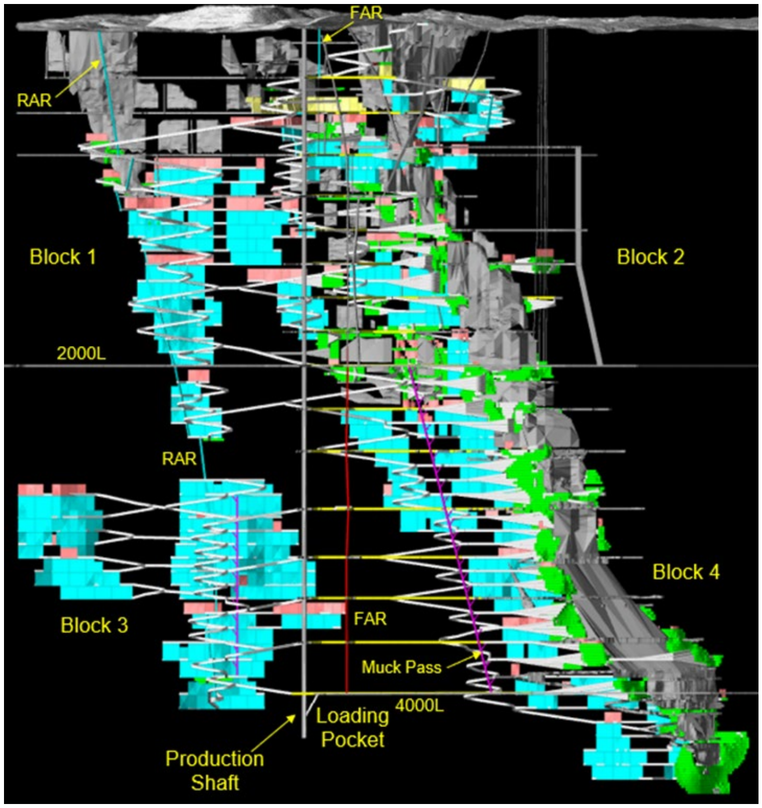
Block 1 and Block 3 are primarily greenfield production areas with Longhole as the primary mining method. Block 2 and Block 4 are brownfield production areas with MCAF used near historic workings and Longhole used in areas not influenced by previous mining. Sublevels will be spaced at 82ft (25 m) vertical intervals, with some variation to accommodate existing development.

There will be three main phases of underground activity.

- Advanced Exploration Program
- Phase 1 Production to 2000L - Mining Block 1 and Block 2 using the Access Ramp
- Phase 2 Production to 4000L - Mining Block 3 and Block 4 using the Production Shaft

The final mine design at the end of Phase 2 is shown in Figure 1-6.

Figure 1-6: Final Underground Mine Design (Isometric view, NTS)



The lateral and vertical development quantities with allowance and overbreak factors applied are summarized by ADEX and mining block in Table 1-2.

Table 1-2: Underground Mined Lateral and Vertical Development Quantities

Item	ADEX (feet)	Block 1 (feet)	Block 2 (feet)	Block 3 (feet)	Block 4 (feet)	Total (feet)
Ramp	4,629	11,875	16,310	20,915	24,224	77,952
Lateral Waste/Crosscuts	1,909	6,859	8,341	10,044	21,752	48,905
Rehab Existing Drifts	0	0	9,375	0	12,378	21,753
Stope Sills	1,701	12,339	16,003	25,056	23,786	78,884
Attack Ramp (cut and fill)	0	3,084	26,560	311	69,192	99,147
Total Lateral Development (ft)	**8,239**	**34,157**	**76,588**	**56,326**	**151,331**	**326,641**
Total Lateral Development (m)	**2,511**	**10,411**	**23,344**	**17,168**	**46,128**	**99,560**
Vertical Development (ft)	**0**	**2,028**	**552**	**3,085**	**1,982**	**7,648**
Vertical Development (m)	**0**	**618**	**168**	**940**	**604**	**2,331**
Total Waste Rock (tons)	**186,738**	**588,923**	**1,495,614**	**827,268**	**3,100,529**	**6,199,072**
Total Waste rock (tonnes)	**169,406**	**534,262**	**1,356,798**	**750,485**	**2,812,753**	**5,623,703**

The diluted and mining recovered production quantities for ADEX and each mining block are summarized in Table 1-3.

Table 1-3: Underground Mine Diluted and Mining Recovered Production Quantities

Production Source	ADEX	Block 1	Block 2	Block 3	Block 4	Total
Development (tons)	21,244	203,566	366,263	419,591	614,439	1,625,103
Longhole Stopes (tons)	0	1,790,524	2,595,443	3,282,108	3,351,941	11,020,016
Cut and Fill Stopes (tons)	0	114,611	1,124,696	9,109	4,045,722	5,294,138
Total Production short tons	**21,244**	**2,108,700**	**4,086,403**	**3,710,808**	**8,012,102**	**17,939,257**
Total Production metric tonnes	**19,272**	**1,912,980**	**3,707,122**	**3,366,388**	**7,268,457**	**16,274,220**
Average NSR / short ton	**$198.61**	**$180.34**	**$159.03**	**$130.57**	**$197.09**	**$172.69**
Average NSR / metric tonne	**$218.93**	**$198.79**	**$175.30**	**$143.93**	**$217.25**	**$190.36**
Average %Ni	0.676	0.519	0.658	0.517	0.736	0.647
Average %Cu	0.709	0.513	0.552	0.395	0.703	0.583
Average %Co	0.020	0.018	0.025	0.018	0.026	0.023
Average Pt (ppm)	1.455	1.281	0.470	0.362	0.632	0.616
Average Pd (ppm)	0.713	1.577	0.380	0.669	0.679	0.715
Average Au (ppm)	0.400	0.773	0.294	0.188	0.410	0.380
Backfill short tons	**0**	**945,735**	**1,846,729**	**1,633,806**	**3,672,303**	**8,098,572**
Backfill metric tonnes	**0**	**857,956**	**1,657,324**	**1,482,164**	**3,331,457**	**7,346,901**

During Phase 1, combined production from Block 1 and Block 2 will ramp up and average approximately 2,400 tons per day. Once the shaft is commissioned for Phase 2 in 2028, overall production will increase to approximately 4,000 tons per day when all blocks are producing before ramping down and ending in

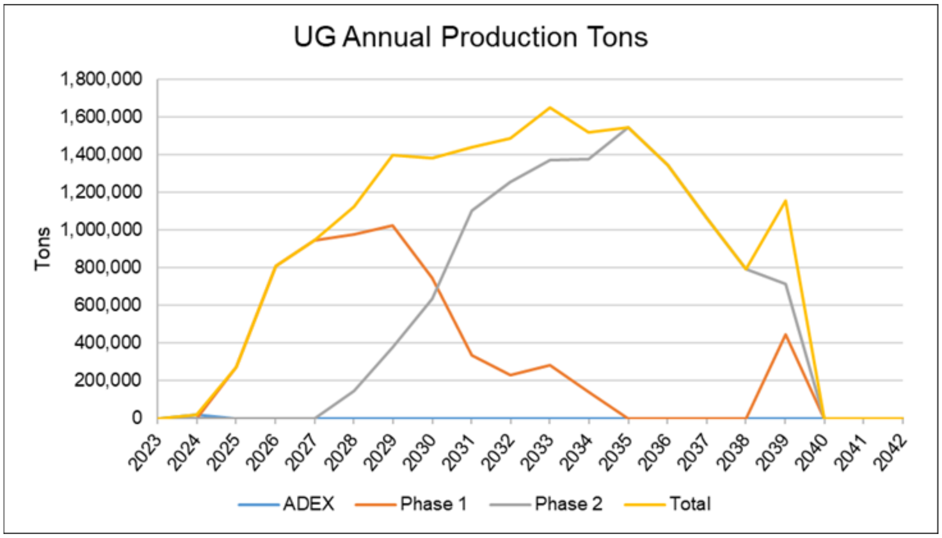
2039. The stopes in Block 2 near the interface with the open pit have been delayed until the end of mine life. The underground production tonnage and average NSR/grade profile (including stope sill development) is summarized in Table 1-4.

Table 1-4: Annual UG Production, NSR, and Grade Profile

Item	2024	2025	2026	2027	2028	2029	2030
Short Tons Metric Tonnes	21,244 19,273	273,458 248,077	806,611 731,745	942,886 855,371	1,124,764 1,020,369	1,397,480 1,267,773	1,379,258 1,251,242
NSR per short ton	$198.61	$138.88	$160.60	$163.74	$174.72	$169.45	$179.62
%Ni	0.676	0.526	0.604	0.556	0.583	0.600	0.656
%Cu	0.709	0.486	0.513	0.567	0.524	0.562	0.598
%Co	0.020	0.020	0.022	0.021	0.021	0.021	0.023
Pt (ppm)	1.455	0.839	0.662	0.644	1.036	0.859	0.661
Pd (ppm)	0.713	0.566	0.742	0.896	1.167	0.940	0.798
Au (ppm)	0.400	0.334	0.358	0.506	0.557	0.525	0.407
Item	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**
Short Tons Metric Tonnes	1,439,258 1,306,200	1,484,791 1,346,980	1,650,235 1,497,068	1,516,953 1,376,157	1,543,573 1,400,306	1,346,399 1,221,432	1,063,595 964,877
NSR per short ton	$210.00	$173.54	$166.27	$170.43	$176.73	$170.13	$173.29
%Ni	0.792	0.663	0.648	0.661	0.681	0.666	0.650
%Cu	0.751	0.550	0.571	0.562	0.590	0.617	0.605
%Co	0.027	0.024	0.023	0.023	0.023	0.022	0.023
Pt (ppm)	0.617	0.564	0.493	0.529	0.551	0.473	0.579
Pd (ppm)	0.760	0.682	0.577	0.635	0.633	0.546	0.637
Au (ppm)	0.413	0.363	0.304	0.320	0.361	0.305	0.332
Item	**2038**	**2039**	**Total**				
Short Tons Metric Tonnes	791,349 717,899	1,156,823 1,049,452	17,939,257 16,274,220				
NSR per short ton	$153.75	$164.26	$172.69				
%Ni	0.574	0.657	0.648				
%Cu	0.519	0.581	0.583				
%Co	0.021	0.023	0.023				
Pt (ppm)	0.457	0.496	0.616				
Pd (ppm)	0.645	0.485	0.715				
Au (ppm)	0.328	0.286	0.380				

The annual underground mine tonnage profile is shown in Figure 1-7.

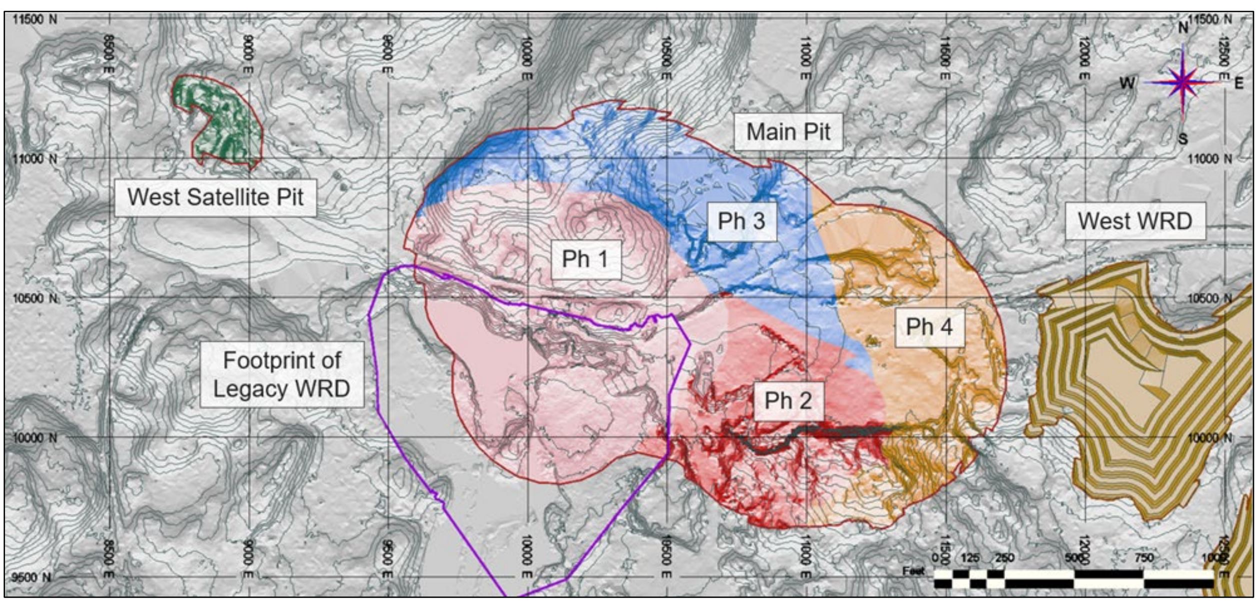
Figure 1-7: Underground Mine Production Tons Profile



1.8.2 Open Pit Mine

The Crean Hill open pit mine plan was established to extract near-surface mineralization. The Crean Hill open pit will be mined using a conventional shovel and truck fleet. The size and geometry of the pit were defined using an economic pit optimization process with the pseudo-flow optimization algorithm. A pit design and phasing were created using the pit shell produced during the optimization process as a guide.

Figure 1-8: Proposed Mining Phases for Crean Hill Open Pit Shown Without the Legacy Waste Rock Dump (WRD)



The pit optimization and pit design were based on the most recent mineral resource model prepared by SGS in 2022.

To determine the optimum economic pit, the pit optimization process for the Crean Hill open pit was performed in two stages. This reason for this was that waste rock would be stored in two separate locations, one close to the proposed pit and one several kilometers away. This two-phased pit shell selection process would allow for the application of two mining unit costs, based on different waste rock hauling costs.

1.8.2.1.1 GEOTECHNICAL DESIGN CRITERIA

No new geotechnical drilling data or information was available for this valuation of the Crean Hill open pit. Stantec assumed geotechnical design parameters that closely resemble the recommendations by Tetra Tech from 2012. The assumed parameters are shown in Table 1-5.

Table 1-5: Pit Slope Angles Assumed for This Study

Geotechnical Domain	Wall Sector Azimuth (°)	Bench Height (m)	Bench Face Angle (°)	Design Overall Slope Angle (OSA) (°)
All Domains - (Norite & MTBS)	0 - 360	5.0	70	38
Old Workings Backfill Material	0 - 360	5.0	37	26

1.8.2.1.2 FINANCIAL PARAMETERS

This economic evaluation of the Crean Hill deposit is premised on the assumption that an ore sales agreement will be entered into with an established processing facility in Sudbury. The details of this agreement, combined with economic values like metal prices and the discount rate were inputs to pit optimization.

To account for revenue, a Net Smelter Return (NSR) value was calculated for every block in the block model, expressed as a value per short ton with a CAD$/T unit. This NSR value was used as a proxy for grade in the optimization process. The NSR value combines several parameters that typically influence the value of the material in a base metal processing plant. The key determinants of the NSR are the metal prices for the metal constituents, the recoveries for each metal, the payable amounts, and treatment and refining costs.

1.8.2.1.3 CUT-OFF VALUE

The cut-off value for the Crean Hill open pit was determined by three factors:

1. Variable processing cost of C$35.38/T.
2. Transport cost from Crean Hill to the processing facility at C$5.00/T.
3. Crushing cost. Mined resource crushing and handling (loading) was estimated as C$2.00/T.

The total of these values yielded a marginal cut-off value of C$42.38/T for the pit.

Except for the waste rock hauling cost, all other determinants of the shell selection process were common to both stages, and the optimal pit shell for each cased was determined by using the Lerchs-Grossman (LG) algorithm as embedded in the Whittle™ software suite.

Utilizing the NSR values in the block model, the Whittle pit optimization software was set up, and several optimization runs were conducted. Table 1-6 outlines the parameters used for the initial optimization runs.

Table 1-6: Pit Optimization Input Parameters

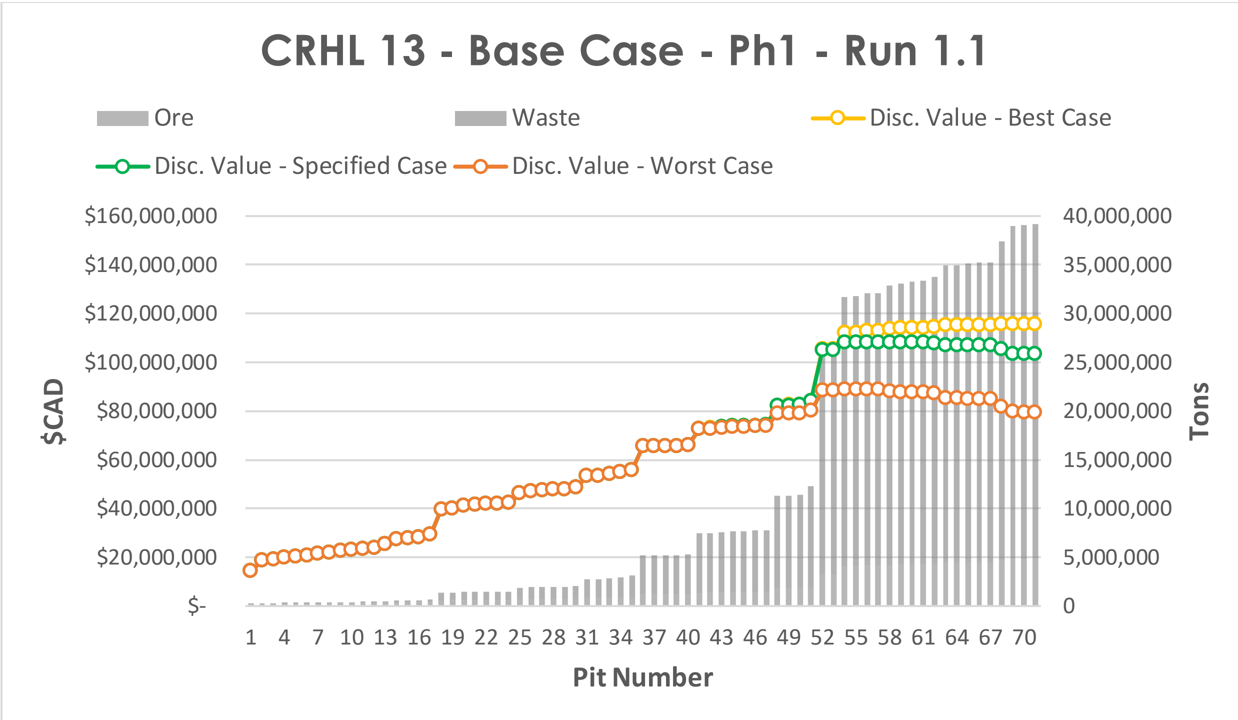
Parameter	Value	
Mining Cost*	$3.00 / ton	$3.37 / ton
Processing Cost	$42.38 / ton	
Overall Pit Slope	38°	
Dilution	5%	
Mining Recovery	95%	
Discount Factor	8%	
Processing Rate**	1.0 MT / year	
Revenue Factor increments	0.10	

** Initial value – revised for design and schedule

The typical output from the Whittle software is a pit-by-pit graph that provides a graphical representation of the incremental values provided by each pit shell. Figure 1-9 shows the results of the first pit optimization run in Whittle™. The secondary vertical axis shows the tons of material contained in the respective pits; and the primary vertical axis shows the discounted value of the respective pit shells.

An important aspect of this output is the ‘steps’ in the graph, which provides insight into the selection of the most optimal pit phases.

Figure 1-9: Whittle Output for the First Optimization Stage



For the second stage of the pit optimization, pit 48 above was used as the topography. The software was re-run and another set of optimization results were obtained.

The ultimate pit will contribute approximately 3.8 million tonnes of mineralized resource to the Crean Hill production profile. See Table 1-7.

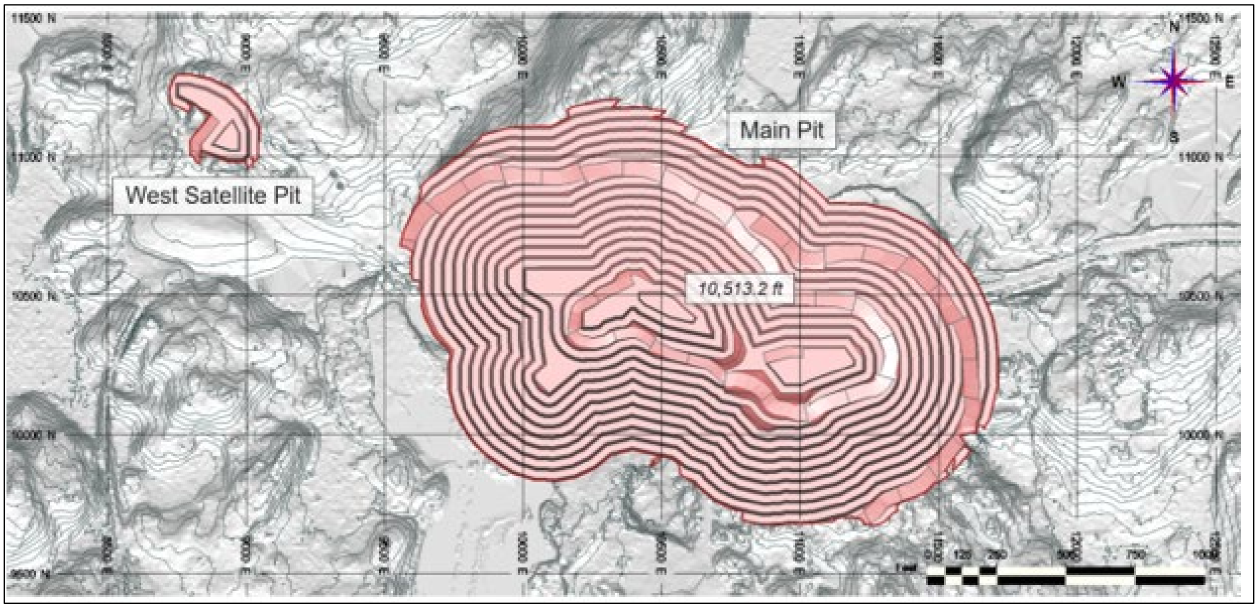
1.8.2.2 PIT DESIGN

A key design criterium for the open pit was the width of the access ramps. This importance of this design aspect was brought about as a result of the relatively small pit chasing resource as deep as economically viable. Several combinations of truck size and road width were considered. A significant problem arose from the fact that wider road widths push out the walls of the pit and significantly increased the strip ratio. The small pit also necessitated several switchbacks which exacerbated the situation.

Eventually the gradient of the haul road was increased to 12% and truck size were reduced to 50t Articulated Dump Trucks (ADT's). This reduced the required road width to 100 feet. This narrower road, combined with the steeper ramp, resulted in less haul road distance, fewer switchbacks and hence required less waste rock removal.

The ultimate pit design is presented in Figure 1-10. The pit was designed to be mined in 32.8 ft (10m) high benches to incorporate wider catch benches without sacrificing mineralized material. A typical catch bench was designed to be 26.2 ft (8m) wide.

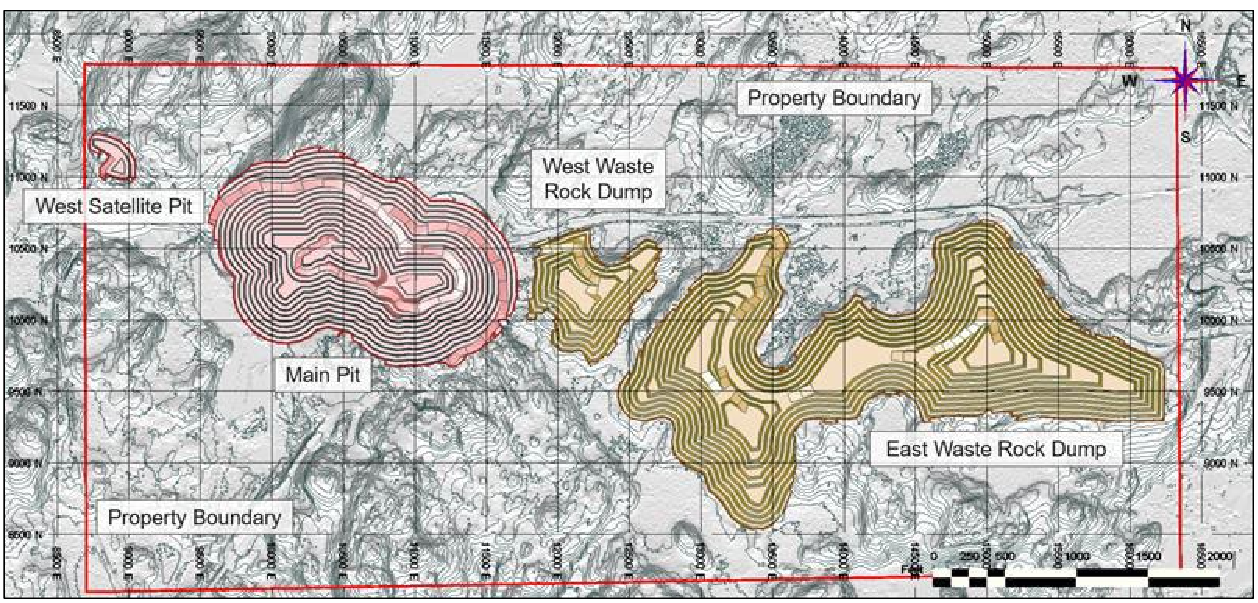
Figure 1-10: The Proposed Crean Hill Open Pit Upon Completion



1.8.2.2.1 WASTE ROCK STORAGE FACILITIES

Two waste rock storage facilities are planned for the Crean Hill open pit. The facility that will be utilized first is situated close to the pit and is shown in Figure 1-11.

Figure 1-11: Crean Hill Waste Rock Storage Facility



This facility was planned to avoid current watercourses and low-lying areas. It will be able to contain approximately 9.6 million short tons of waste rock. The remainder of the waste rock will be stored in a historic open pit approximately 2 kilometers from the Crean Hill open pit or at another area on an adjacent property.

The Crean Hill pit yielded the key results summarized in Table 1-7.

Table 1-7: Crean Hill Open Pit Key Numbers

Material Tonnages	Value	Units
Potentially Economic Material[1]	3.83	Million tonnes
Legacy Waste	4.28	Million tonnes
Pit Waste	29.35	Million tonnes
Total	33.18	Million tonnes
Strip Ratio	8.78	: 1
Grades		
Dilute NSR Value	131.05	$CAD/tonne
Dilute Nickel Grade	0.53	%
Dilute Copper Grade	0.41	%
Dilute Cobalt Grade	0.019	%
Dilute Platinum Grade	0.36	ppm
Dilute Palladium Grade	0.22	ppm
Dilute Gold Grade	0.17	ppm

Notes:

1. Potentially economic material tonnage and grade values are diluted at 5%
2. A mining recovery rate of 95% was applied.
3. Ore tonnage were determined with a dilute NSR cut-off value of $42.38/Ton

1.8.2.3 END-OF-PERIOD MAPS

1.8.2.3.1 PIT DEVELOPMENT STAGES

The following drawings show the Crean Hill open pit in different stages of development.

The figures below show the progression of open pit mining. Figures do not show construction of surface infrastructure, waste rock dump construction, or any other features external to the pit. Elevation contours of topography and the open pit are presented in 5 ft (1.5 m) and 16.4 ft (5 m) intervals respectively.

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Figure 1-12: Original Topography with Pit (Pre-Mining)

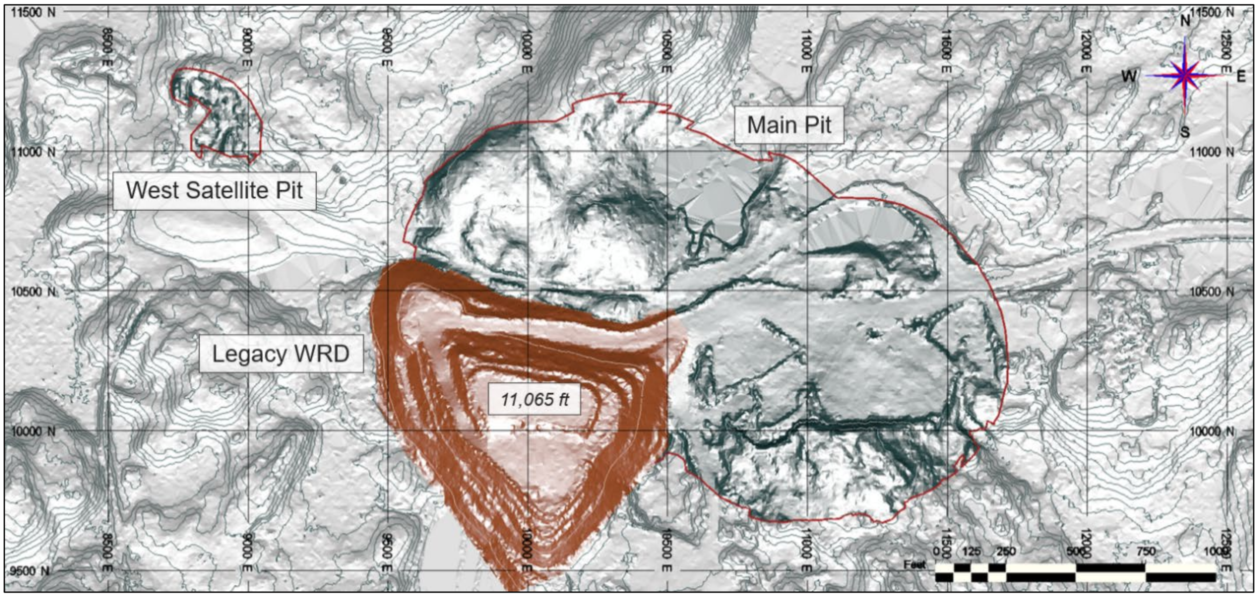


Figure 1-13: End of Q2 2025

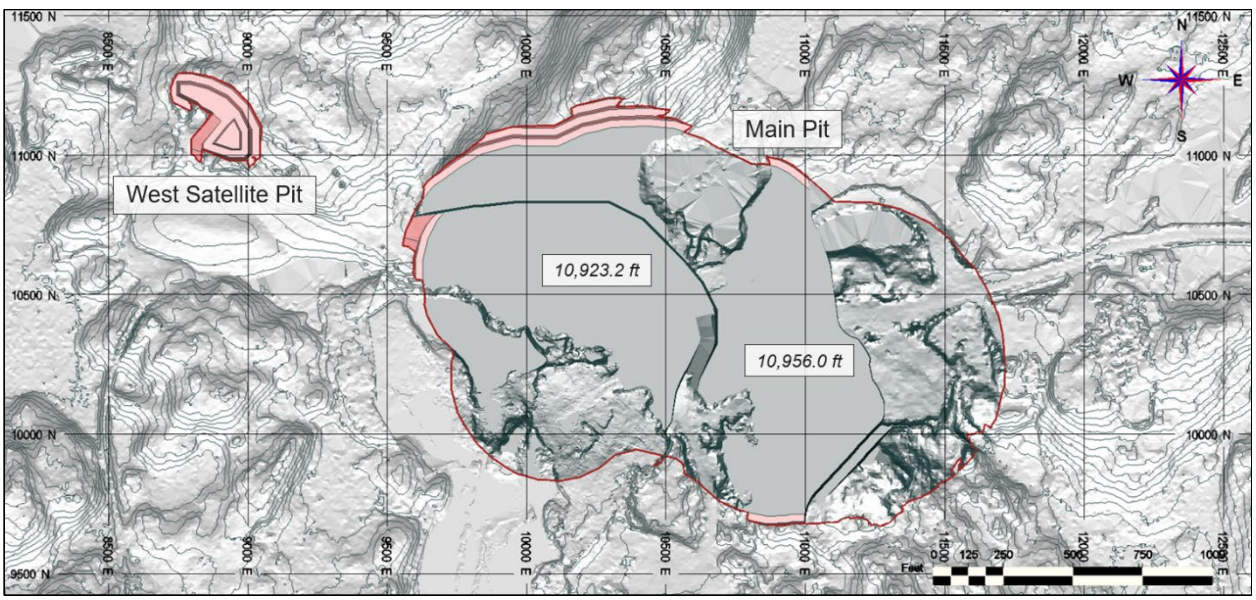


Figure 1-14: End of Q4 2025

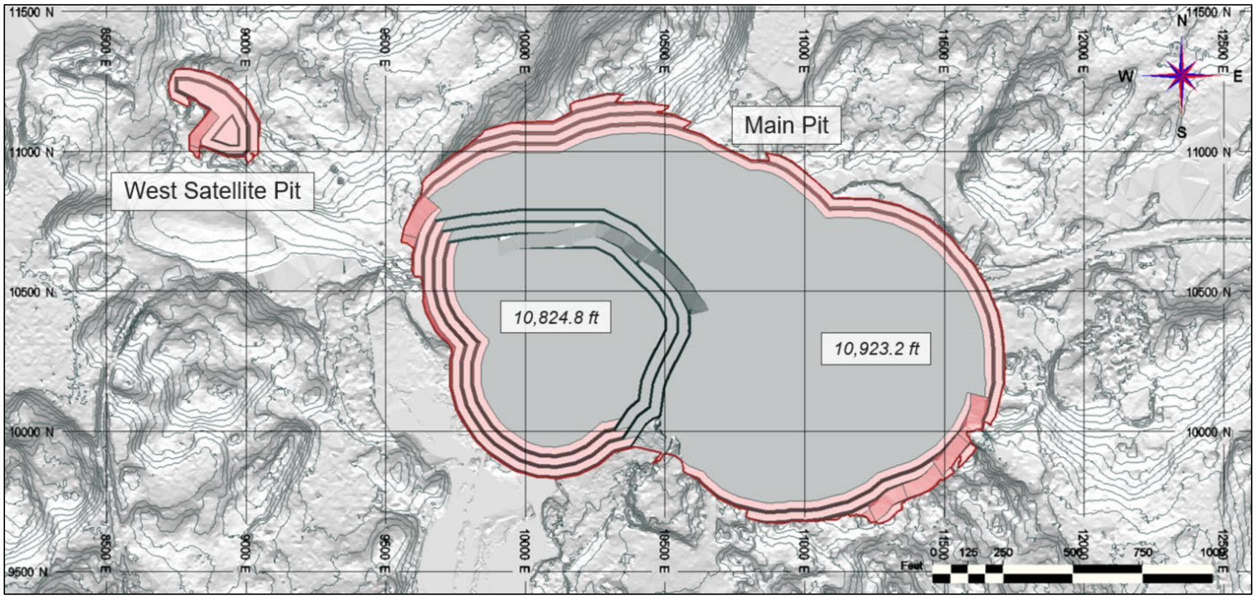


Figure 1-15: End of Q2 2026

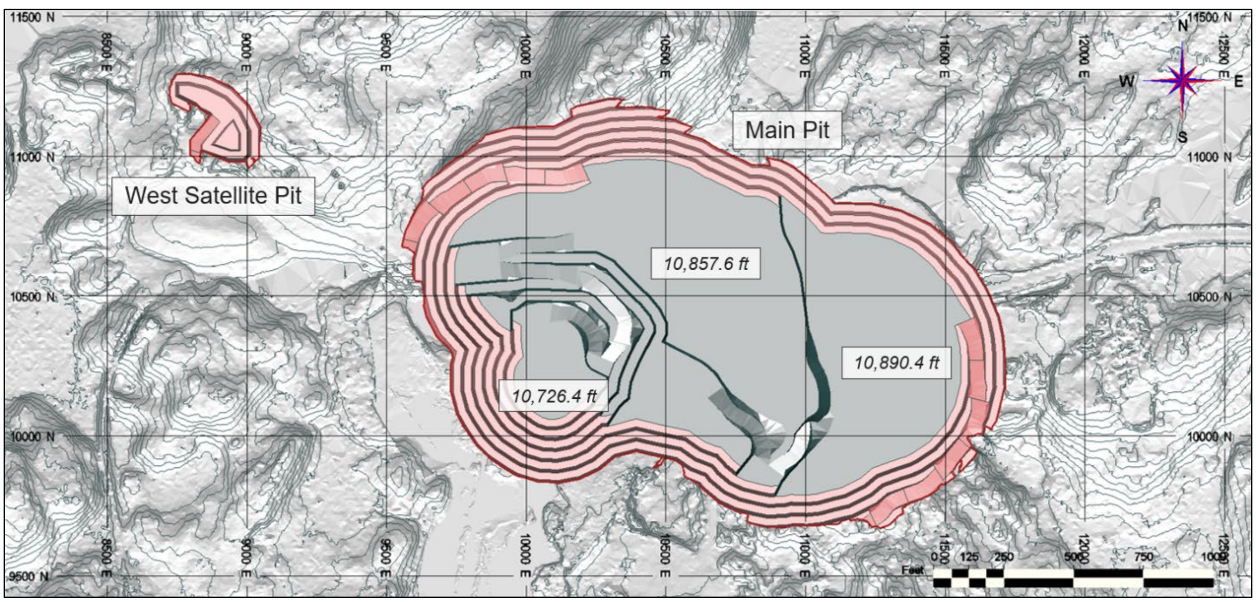


Figure 1-16: End of Q4 2026

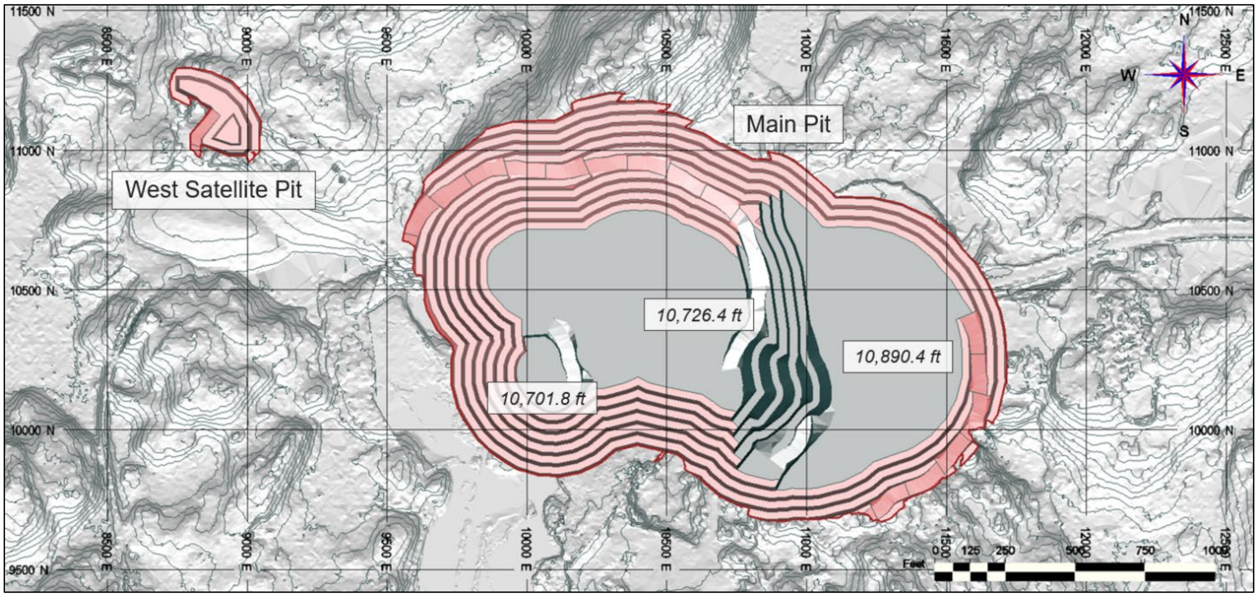


Figure 1-17: End of 2027

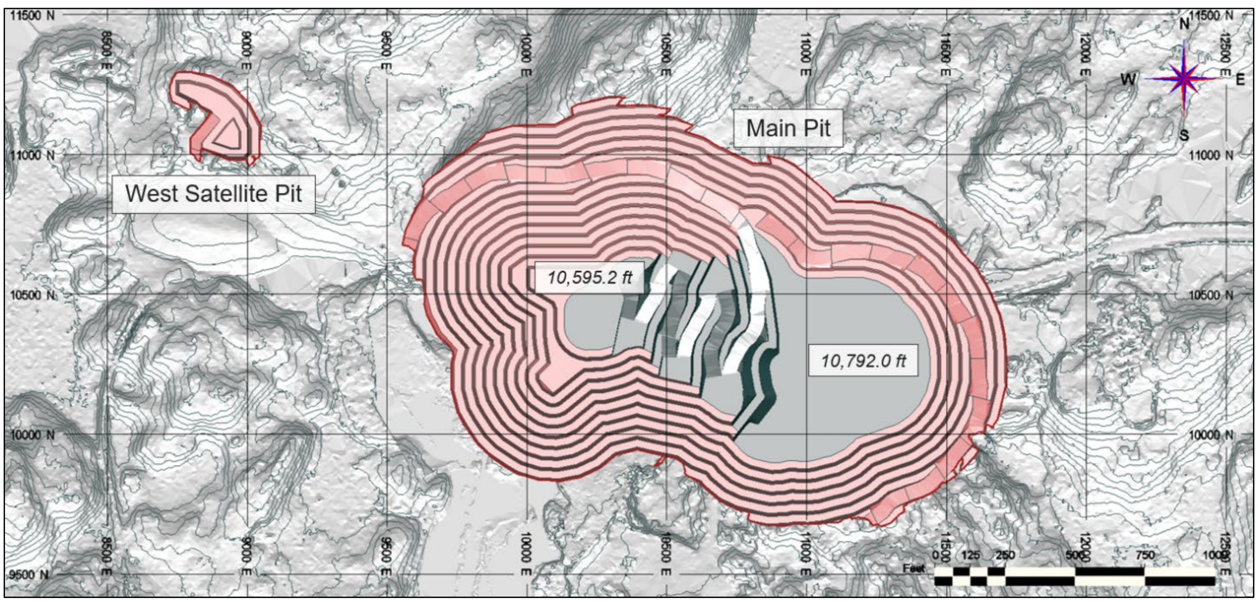


Figure 1-18: End of 2028

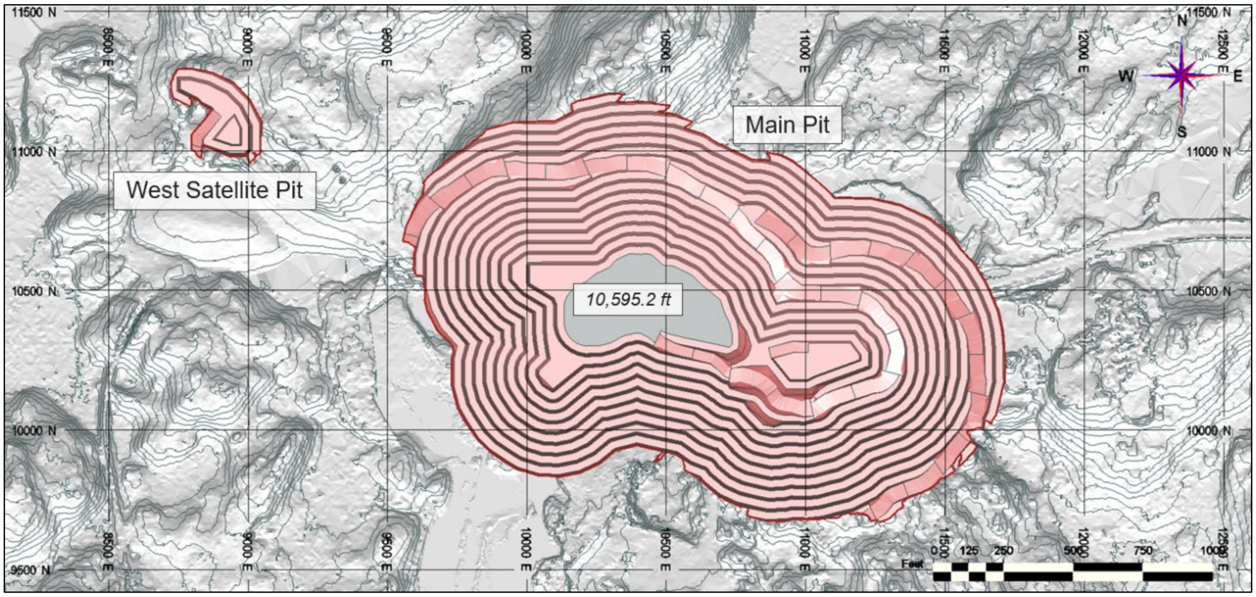


Figure 1-19: End of 2029

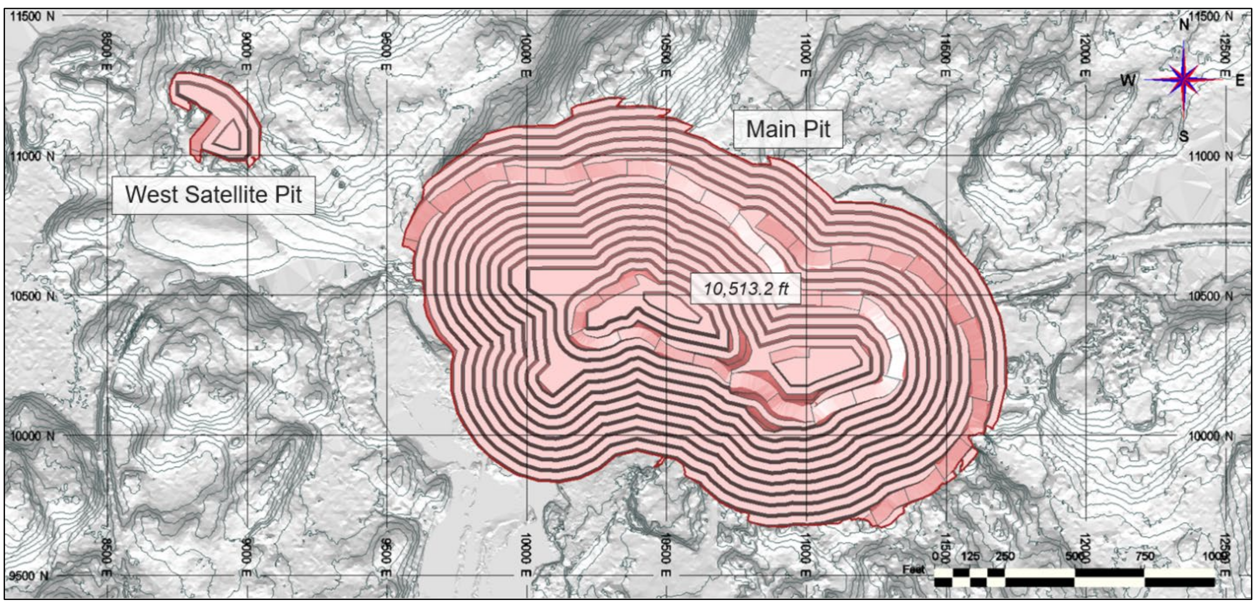
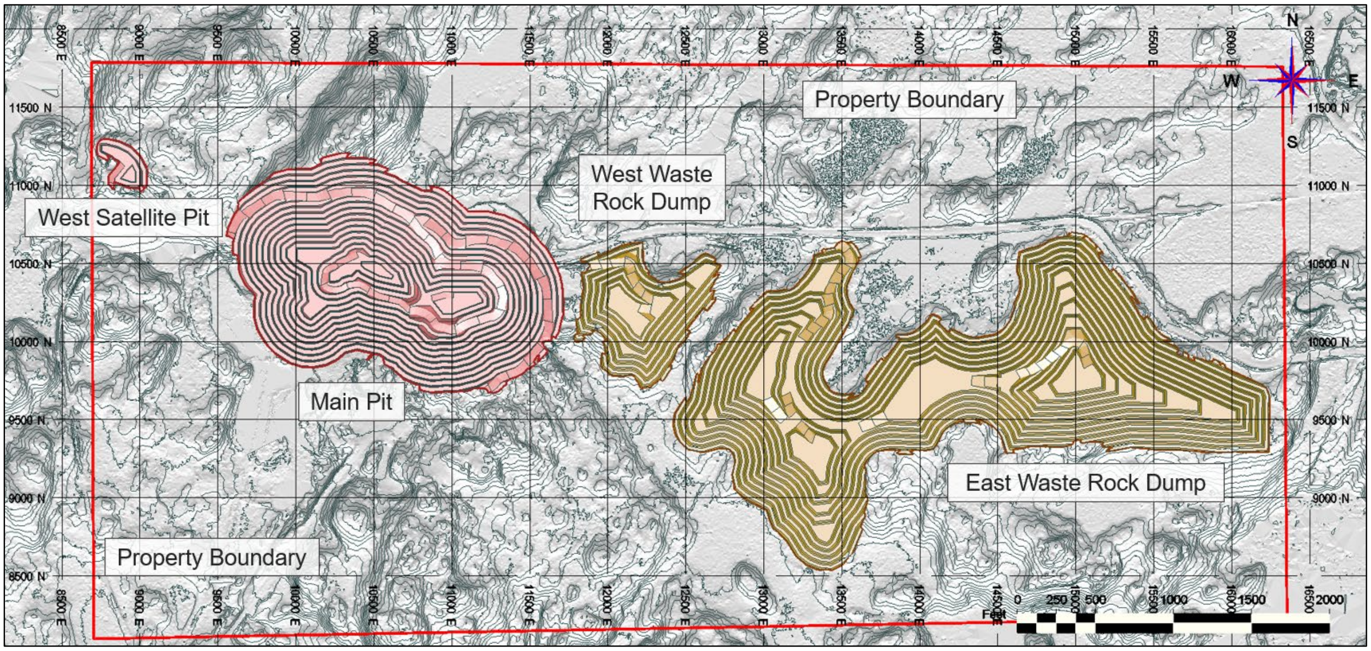


Figure 1-20: Crean Hill Waste Rock Storage Facility (WRSF) and End-of Open Pit Life



1.8.2.4 MINING EQUIPMENT

Two design criteria were considered in the selection of the fleet. These were:

1. Drilling, blasting, loading and hauling on average 17,200 tons of material per day out of the pit.
2. Haul trucks able to negotiate a 12% ramp in all weather conditions.

To satisfy the second requirement, an articulated dump truck (ADT) with a 50 metric tonne payload was selected as the prime earth mover. Although these trucks have a higher unit operating cost, they are all-wheel driven and able to maintain traction in muddy and wet conditions. The narrow width of these trucks is important for the pit as designed, allowing for ramp widths of 50 feet. The width of the truck over the widest tire option is 12 feet.

To match the 50-ton Bell ADT's the CAT390D loader was selected as the primary loader. It can load the 50-ton hauler in 4 passes. To round out the equipment fleet the following equipment would also be required:

- One medium size wheel loader (Front-End-Loader) (FEL)
- Two track dozers (CAT D8 size) – one in the pit and one at the rock storage facility
- One medium size road grader
- A tire dozer – for cleaning loading areas and road maintenance
- A small fleet of ancillary equipment

1.8.2.5 MINING CAPITAL

Since the open pit life is limited to 3 years, a contract mining operation is planned. Therefore, capital for the open pit component of the Crean Hill mine is limited to dewatering infrastructure and a capital contribution to the crushing infrastructure.

1.9 RECOVERY METHODS

The recovery approach discussed in this report is based on indicative terms for an ore sale agreement with a local mining company. Detailed metallurgical testing of the 109 FW and general knowledge of the behaviour of contact deposits within the Sudbury basin indicate that processing these resources will yield positive returns. Before entering into an ore sale agreement with a local mining company, mineralogical and metallurgical testing will be performed on samples of the materials that will be processed, according to the local mining company's selection and testing protocols. Based on these tests, the recovery of metals to concentrate at target concentrate grade will be confirmed and a sales agreement can be finalized.

1.10 INFRASTRUCTURE

The Crean Hill Project is a former operating mine site with existing year-round road access, nearby electrical grid power, and available water sources. The Project will have access to mining-industry leading service providers, suppliers and supply chains, and labour markets available in Sudbury and surrounding communities.

1.11 ENVIRONMENTAL

The climate in the Property area is characterized by moderately long, cold winters and shorter, warm summers, as is typical of continental conditions. The area experiences a wide variation in temperature throughout the year. In winter months, the temperature generally drops below -20°C for extended periods. In the summer, the maximum daily temperature may reach over 25°C for extended periods. The daily mean temperature typically falls below freezing from December through March. Precipitation in the region is moderate and is distributed evenly throughout the year, with only minor seasonal trends. However, the wettest months generally occur from May to October. Canadian Climate Normals (1981 to 2010) for the Sudbury airport estimate average annual total precipitation at 903 mm, with 676 mm falling as rain and 228 mm (water equivalent) falling as to snow.

The topography at the Property is rugged, with rock knobs representing the dominant bedrock landform in the area. These knobs are often bare or covered with a metre or less of boulder-strewn sandy till, thickening between the highs to between approximately 3 m and 5 m. Slopes are generally steep and complex, and relief ranges between 15 m and 60 m. The exposed rock knobs themselves are well drained. Organic deposits that are often found confined between outcrops and are generally observed to be low lying and wet. Drainage to the Vermillion River and several smaller creeks is poor, and as a result, small swamps and marshes are numerous. The Property is contained entirely within the Vermillion River watershed. The Vermillion River eventually discharges to the Spanish River, which drains to the North Channel of Lake Huron. The majority of the Property drains to three (3) separate sub-watersheds that flow into the Vermillion River including Fairbanks Creek Watershed situated to the west and northwest of the Property, Northeast Watershed covering the east end of the Property with drainage flowing eastward through a series of wetlands and beaver ponds to the Vermillion River and the Monk Lake Watershed where most of the former Crean Hill Mine site is situated and Monk Lake is used as a water treatment facility to treat runoff and dewatering from mining activities at the Property.

As presented in the Crean Hill Mine Closure Plan (Vale, 2022), surface water quality has shown a wide range of concentrations for dissolved metals, as well as sulphate and pH values, generally demonstrating effects from natural mineralization as well as historic mining activities. Groundwater flow systems have been identified and characterized around the Property, as described in the Crean Hill Mine Closure Plan (Vale, 2022). Groundwater flows away from the site towards the Vermillion River, which is located approximately 2 km to the southeast. Groundwater quality in overburden has shown a wide range of concentrations for dissolved metals, as well as sulphate and pH values, generally demonstrating effects from natural mineralization as well as historic mining activities. Groundwater quality for the parameters of interest for bedrock flow system at the Property have either remained the same or have improved over the 2011 to 2018 monitoring period, as documented in the Crean Hill Mine Closure Plan (Vale, 2022).

Three (3) main soil types have been identified on the Property. The soils, according to maps produced by the Soil Survey of Canada, include Rockland, Monteagle, and Baldwin (Soil Survey of Canada, 1983).

Given the abundant mineralization at the Property, the mitigation of chemical instability issues (i.e., acid generation, metal leaching) has been the focus of the Water Quality Management Plan for the Property that is being implemented by Vale Canda Limited (Vale). The Water Quality Management Plan is intended to help Vale make proactive and informed decisions on the management of water quality related environmental risks arising from their mining operations. The management plan establishes a hazard screening and risk evaluation process to assess water quality data for potentially affected water bodies. This process establishes an approach to evaluate and prioritize environmental risks and provides the means to establish remediation priorities and then develop site-specific action plans for mitigation.

The evaluation of the potential for biological impacts considers metal concentrations in water, the bioavailability of the metals, the presence of substances known to reduce toxicity including underlying geochemistry of the receiver (levels of calcium, magnesium and dissolved organic carbon, etc.), biotic factors such as acclimation and adaptation, physical setting, and size and location of the receiver. The evaluation of social impacts includes factors such as visibility, potential for health risks, impact on use or enjoyment of the water resource, proximity to urban areas, and proximity to areas of importance to Indigenous communities.

Aquatic resource inventory work has been on-going at the Property in accordance with regulatory requirements including the Metal and Diamond Mining Effluent Regulations (“MDMER”). Environment and Climate Change Canada’s Environmental Effects Monitoring (“EEM”) guidance documents have been followed for recent studies. Despite concentrations of various metals in sediment being elevated above recognized thresholds due to naturally occurring mineralization and historic mining activities, the first two EEM studies found no effects within the benthic invertebrate communities from effluent discharge. As part of ongoing EEM work, fish sampling studies have been undertaken at the Property in 2005, 2007 and 2012.

The 2012 EEM study replicated the fish population study using northern redbelly dace and creek chub, as fathead minnow were unable to be caught in sufficient numbers. The fish capture work identified the presence of white sucker in the receiving environment as well as a variety of small, bodied fish (i.e., fathead minnow, pearl dace, brook stickleback, Iowa darter, finescale dace, creek chub, northern redbelly dace, brow bullhead, central mudminnow, brassy minnow and common shiner).

Background flora and fauna studies for the Property were not conducted prior to initial development by The Canadian Copper Company in 1905. A site characterization for the Property was conducted in 1993 to inventory and sample foliage from the existing vegetation. As documented in the Crean Hill Mine Closure Plan (Vale, 2022), the levels of metals in the vegetation sampled from around the former mining sites were observed to be slightly elevated.

Animal life at the Property has not been surveyed recently. However, wildlife species such as moose, bear, deer, ruffed grouse, ducks, otter, beaver, and muskrat are common, based on observations by personnel at the site and knowledge from trappers in the area. Presently, Magna is not aware of any Species at Risk or habitat features at the Property that warrant consideration under Ontario's Endangered Species Act.

Culturally sensitive areas and areas with a high potential to host an archaeological or cultural heritage value have not been defined to date. Going forward, Magna will consult the proximal Indigenous communities and review available electronic databases (Natural Heritage Information Center, etc.) to identify any areas of cultural or heritage significance.

1.12 MARKETS

No project-specific marketing studies were undertaken for the PEA. The PEA considers the sale of mined mineralized resource from the Crean Hill mine to a third-party mill within trucking distance in the surrounding area.

1.13 CAPITAL AND OPERATING COSTS

The cost estimate was prepared to a Class 5 as defined by AACE International (The Association for the Advancement of Cost Engineering) with an approximate -20 to -30% / +30 to +100% accuracy. All costs in the estimate are reported in Quarter Two (Q2) 2023 Canadian dollars. The costs have been separated into four main cost classifications.

- Advanced Exploration (ADEX)
- Project Capital
- Sustaining Capital
- Operating

The costs associated with the Advanced Exploration (ADEX) program are summarized in Table 1-8.

Table 1-8: Estimated Advanced Exploration Costs

Item	Cost (CAD millions)
ADEX Direct Capital Costs	$20.18
ADEX Indirect Costs	$4.04
ADEX Contingency	$8.47
ADEX Total Capital Costs	**$32.69**
Operating Costs during ADEX	**$15.68**

The capital and operating costs for the project period and operating period are summarized in Table 1-9.

Table 1-9: Estimated Project Period and Operating Period Costs

Item	Underground (CAD millions)	Open Pit (CAD millions)	Combined (CAD millions)
Project Period Direct Capital Costs	$34.63	$15.42	$50.06
Project Period Indirect Costs	$6.93	$3.08	$10.01
Project Period Contingency	$14.55	$6.48	$21.02
Sustaining Capital Costs	$247.16	$1.70	$248.86
Total Capital Costs	**$303.27**	**$26.69**	**$329.96**
Operating Costs during Project Period	$13.33	$21.57	$34.90
Operating Cost during Operating Period	$1,921.02	$371.70	$2,292.73
Total Operating Costs	**$1,934.35**	**$393.27**	**$2,327.62**
Closure Costs	**$5.85**	**$11.24**	**$17.09**

1.14 FINANCIAL ANALYSIS

The Crean Hill Project combined Underground and Open Pit mining operation demonstrates a potential post-tax net present value (NPV) (discounted 8%) of $230.4 million, IRR of 23.4%, and 3.3-year payback period (from the start of the project period) during a 15-year mine life. The estimated life of mine taxes total $163.3 million and reflect current provincial and federal legislation. There is a 3% NSR royalty on the property.

The cash flow, NPV, and IRR results are summarized in Table 1-10.

Table 1-10: Crean Hill Project Cash Financial Summary

Item	Underground (CAD millions)	Open Pit (CAD millions)	Combined (CAD millions)
Project Period Direct Capital Costs	$34.63	$15.42	$50.06
Project Period Indirect Costs	$6.93	$3.08	$10.01
Project Period Contingency	$14.55	$6.48	$21.02
Sustaining Capital Costs	$247.16	$1.70	$248.86
Total Capital Costs	**$303.27**	**$26.69**	**$329.96**
Operating Costs during Project Period	$13.33	$21.57	$34.90
Operating Cost during Operating Period	$1,921.02	$371.70	$2,292.73
Total Operating Costs	**$1,934.35**	**$393.27**	**$2,327.62**
Closure Costs	**$5.85**	**$11.24**	**$17.09**
Royalties	**$90.42**	**$14.42**	**$104.84**
Pre-Tax NPV8% (includes ADEX)	**$251.99**	**$38.44**	**$290.43**
Pre-Tax IRR (includes ADEX)	**22.43%**	**37.82%**	**23.91%**
Taxes	**$146.86**	**$16.69**	**$163.29**
Post-Tax NPV8% (includes ADEX)	**$203.02**	**$26.12**	**$230.44**
Post-Tax IRR (includes ADEX)	**22.39%**	**29.62%**	**23.37%**

1.15 CONCLUSIONS

1.15.1 Geology and Exploration

The general requirement that all mineral resources have “reasonable prospects for eventual economic extraction” implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade, taking into account extraction scenarios and processing recoveries. To meet this requirement, the author considers that the Crean Hill deposit mineralization is amenable for open pit and underground extraction.

1.15.2 Mining

Considering the inferred and indicated resource material in the resource model, the Crean Hill Project shows potential for a combined Underground and Open Pit mining operation that could produce up to 20.1 million metric tonnes over a nominal 15-year mine life at an average NSR of $179.07 per tonne with a third-party toll-milling arrangement. The potential production is summarized in Table 1-11.

Table 1-11: Crean Hill PEA Potential Production

Item	Underground	Open Pit	Combined
Tonnes Mined	16,274,220	3,828,385	20,102,605
Average NSR $/tonne	$190.36	$131.05	$179.07
%Ni	0.647	0.525	0.624
%Cu	0.583	0.408	0.549
%Co	0.023	0.019	0.022
Pt (grams per tonne)	0.616	0.365	0.569
Pd (grams per tonne)	0.715	0.225	0.621
Au (grams per tonne)	0.380	0.173	0.341

1.15.3 Metallurgy and Processing

The metallurgical testing of the FW zones supports the recoveries used in this report. The material is similar to other Sudbury Basin material and can be processed by a local mining company as an ore sale agreement based on the indicative terms provided.

1.15.4 Costs and Financial Model

The Crean Hill Project combined Underground and Open Pit mining operation demonstrates a potential post-tax NPV (discounted 8%) of $230.4 million, IRR of 23.4%, and 3.3-year project capital payback period (from the start of the project period) during a 15-year mine life.

1.16 RECOMMENDATIONS

Based on the results of the Preliminary Economic Assessment, the QPs recommend the Crean Hill Project advances to a Pre-Feasibility Study (PFS) with an updated resource model that includes additional diamond drilling information. The purpose of the PFS is to advance the engineering of the mine designs and infrastructure designs, investigate unknowns and assumptions, update the cost estimate to AACE Class 4, and support a mineral reserve estimate for the Project Recommendations and estimated budgets to complete the work are summarized in Table 1-12.

Table 1-12: Recommendations for Future Work

<table>
<tr><th>Future Work</th><th>PFS Estimated Cost (CAD)</th></tr>
<tr><td>Geology
Diamond Drilling (12,000m to 15,000m).
Revised mineral resource model.
Revised mineral resource estimation.
Surface Bulk Sample.</td><td>$3,425,000
To
$4,225,000</td></tr>
<tr><td>Environmental
• Continue to stay apprised of Vale Crean Hill/Ellen pit closure activities.
• Confirm future water treatment capacity at the Crean Hill treatment plant to accept open pit and underground working dewatering and mine redevelopment runoff.
• Maintain minimum environmental setbacks/buffers from waterbodies/watercourses.
• Avoid deposition of mine waste in waters frequented by fish.
• Confirm the fish habitat status of on-site waterbodies.
• Conduct wildlife and terrestrial vegetation surveys throughout the property.
• Confirm the production history that will establish the threshold for triggering a federal impact assessment and consider this threshold in the mine production rate.
• Continue geochemical characterization of mineralized material, waste rock and overburden.
• Continue engagement with indigenous stakeholders and the local community.
• Commence regulatory engagement towards mine re-opening.</td><td>$85,000</td></tr>
<tr><td>Processing
• Evaluate samples from untested zones.
• Evaluate blending to enhance precious metal recovery.
• Evaluate impact of gravity separation on coarse primary grind.
• Evaluate ore sorting.
Evaluate processing through a future Shakespeare Mill.</td><td>$350,000</td></tr>
<tr><td>Underground and Open Pit Mine Design and Infrastructure
• Geomechanical studies and investigations.
• Investigate the condition of the existing shaft.
• Complete desktop trade-off studies.
• Advance mine designs and schedules.
• Reserves estimate.
• Advance infrastructure designs.
• Cost estimates and financial analysis.
• NI 43-101 Technical Report</td><td>$1,560,000
to
$2,520,000</td></tr>
</table>

2 INTRODUCTION

2.1 ISSUER

Magna Mining Inc. (Magna) is a Canadian exploration and development company based in Sudbury, Ontario Canada, and is publicly listed on the Toronto Stock Exchange (TSXV: NICU). Magna is Sudbury focused, with a vision of bringing new and past producing mines into production, including the development of Magna's 100% owned Crean Hill Ni-Cu-platinum group metals project (Crean Hill Project or the Project) near Sudbury.

Magna retained independent consultants to prepare this technical report for a Preliminary Economic Assessment (PEA) on the Crean Hill Project.

2.2 PURPOSE OF THIS TECHNICAL REPORT

This technical report provides a preliminary assessment of the potential economics for the Crean Hill Project.

The Qualified Persons (QPs) authoring this report caution that this PEA is preliminary in nature and includes inferred resources that are too geologically speculative to have economic considerations applied to them and which cannot be converted to a viable mineral reserve. There is no certainty that the indicative economic value presented in the PEA will be realized, and it should only be treated as an initial analysis of the potential viability of the Crean Hill Project mineral resources.

2.3 EFFECTIVE DATE

The effective date of this technical report is 31 July 2023.

2.4 SOURCES OF INFORMATION

The PEA study team has used information from a digital database and internal technical reports provided by Magna, as well as other public information. The Project's most recent technical report was prepared for Magna Mining Inc. by SGS Canada Inc. in 2022 and is titled "Technical Report on The Mineral Resource Estimate for The Denison Ni-Cu-PGE Sulphide Deposit, Denison Project, Sudbury, Ontario Canada" (dated 14 December 2022, effective 19 August 2022). This report was made public.

Other key documents used as information sources include the following:

- The Project was the subject of a technical report by SRK in 2020 titled "Preliminary Economic Assessment for the Denison Base Metal Project, Final Report." The report was prepared for Lonmin Canada Inc., issued December 2020, and effective 04 December 2020. This report was an internal document and was not made public.
- The Project was subject of a technical report by WSP in 2020 titled "Denison Project Resource Review, Denison Twp., Sudbury District." It was prepared for Lonmin Canada Inc., issued 26 November 2020, and effective 29 September 2020. This report was an internal document.

- The Project was the subject of a technical report by Golder Associates in 2015 titled "Denison Pit Project Geotechnical Review." It was prepared for Lonmin Canada Inc. and issued 14 April 2014. This report was an internal document.
- The Project was the subject of a Closure Plan Amendment by CVRD Inco Limited in 2015 titled "Crean Hill Closure Plan Part 1 of 2 Final." It was issued November 2007. This closure plan is a public document.

2.5 QUALIFIED PERSONS

This technical report is authored by Stantec Consulting Ltd. The following QPs have contributed to sections of this technical report related to their areas of expertise. Through their education, membership to a recognized professional association, and relevant work experience, they are all independent QPs as defined by NI 43-101.

- Michael Murphy, P.Eng. – Stantec Consulting Ltd. (Underground Mining, Capital and Operating Costs, Economic Analysis).

- Christiaan Terblanche, P.Eng. MBA – Stantec Consulting Ltd. (Open Pit Mining, Capital and Operating Costs, Economic Analysis).

- Sheldon Smith, P.Eng. – Stantec (Environmental).

- Allan Armitage, Ph.D., P.Geo. – SGS Canada Inc. (Geology, Exploration, Drilling, Sample Preparation Drilling and Security, Data Verification, Mineral Resource Estimates).

- Gordon Marrs, P.Eng. – XPS – Expert Process Solutions (Mineral Processing and Metallurgical Testing)

The QPs' responsibilities are summarized in Table 2-1.

Table 2-1: QP Responsibilities

Company	QP	Site Visit	Responsibility
Stantec	Michael Murphy	09 December 2022	Underground Mining Cost Estimate / Financial Analysis Section 1-6, 15, 19 Parts of Sections 16, 18, 21, 22, 23, 24, 25, 26, 27
Stantec	Christiaan Terblanche	09 December 2022	Open Pit Mining Cost Estimate / Financial Analysis Parts of Sections 1, 16, 18, 21, 22, 23, 24, 25, 26, 27
Stantec	Sheldon Smith	26 January 2023	Environment and Permitting Parts of Section 1, 25, 26, 27 Section 20
SGS	Allan Armitage	25-26 May 2022	Geology and Mineral Resources Section 7-12, 14 Parts of Section 1, 25, 26, 27
XPS	Gordon Marrs	No Visit	Mineral Processing Section 13, 17 Parts of Section 1, 25, 26, 27

2.6 SITE VISITS AND PERSONAL INSPECTIONS

2.6.1 Geology Qualified Person

Allan Armitage visited the Project site on 25 May 2022. Allan was accompanied by Jason Jessup, CEO and Director of Magna and David King Sr. V.P., Technical Services of Magna. The site visit focused on several outcrops, reviewing the geology and various mineralization styles, rock sample and channel sample locations, and recent and historical drill sites.

On 26 May 2022, accompanied by David King, Allan Armitage visited the Project's core storage facility in Sudbury. Allan examined several selected mineralized core intervals from recent diamond drill holes from the Project, reviewed assay certificates, and compared them against the drill core mineralized zones. All core boxes were properly labelled and stored in core racks. Sample numbers for recent drill holes were written on the core and it was possible to validate sample intervals and confirm the presence of mineralization in certain half-core samples from the mineralized zones. At the time of the site visit, there was no active exploration or mining activities on the Property and Magna had yet to commence exploration on the Property.

Allan was able to become familiar with conditions on the Property and to observe and gain an understanding of the geology and various mineralization styles present. Allan was also able to verify the work done and review and recommend an appropriate exploration or development program to Magna.

2.6.2 Mining Qualified Person

Michael Murphy and Christiaan Terblanche visited the Project site on 09 December 2022. Michael and Christiaan were accompanied by David King Sr. V.P., Technical Services of Magna and Andrew Sullivan, Stantec Project Manager. The site visit focused on several outcrops, reviewing the geology and various mineralization styles, as well as the historic capped workings, waste rock piles, potential portal locations, and existing infrastructure.

2.6.3 Environment Qualified Person

Sheldon Smith visited the Project site on 26 January 2023. Sheldon was accompanied by Derek Teevan of Magna. The site visit focused on the site layout as well as the existing water treatment system being used to manage the site runoff and overflow water from the historic underground workings.

2.6.4 Processing Qualified Person

Gordon Marrs did not conduct a personal inspection of the site.

3 RELIANCE ON OTHER EXPERTS

The Qualified Person's opinions, estimates, and conclusions in this Preliminary Economic Assessment are based on information available at the time of preparation of this technical report, including data, reports, and other information supplied by Magna Mining Inc. (Magna). Where information was not available, reasonable assumptions and qualifications were made as described in this report.

3.1 OWNERSHIP, MINERAL TENURE, AND SURFACE RIGHTS

Ownership information was provided by Magna, and this has been relied on by the QPs. Magna has indicated that all the mining patents on the property are in good standing. The QPs express no opinion regarding the ownership status of the property. Ownership information is discussed in Item 4.

3.2 HISTORIC MINE WORKINGS

Information on the historic mine workings was provided by Magna in the form of electronic data files. The completeness and accuracy of the historic mine workings is used in the mineral resource estimate discussed in Item 14 and the Underground and Open Pit mine design discussed in Item 16.

3.3 ENVIRONMENTAL AND PERMITTING

Information on environmental and permitting was provided by Magna. The information was relied on by Stantec and discussed in Item 20.

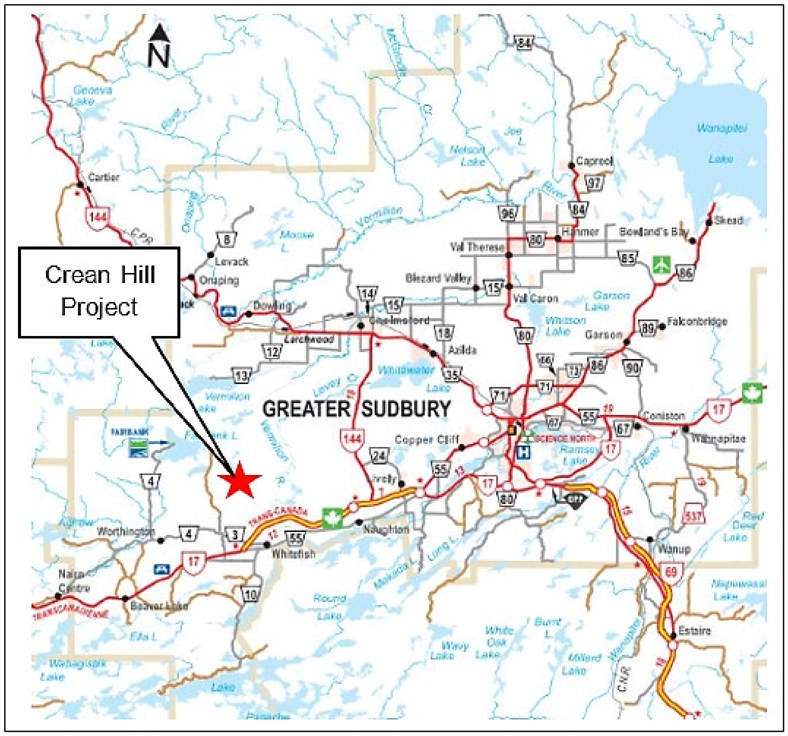
4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Crean Hill Project property is located in Denison Township within the City of Greater Sudbury, Ontario, Canada, approximately 30 km southwest of downtown Sudbury (refer to Figure 4-1 and Figure 4-2). The property is centered at approximately 46° 25.8' N latitude, 81° 21.1' W longitude (473,000 m E, 5,141,800 m N in NAD83 UTM Zone 17N).

Figure 4-1: Project Location on Map of Ontario

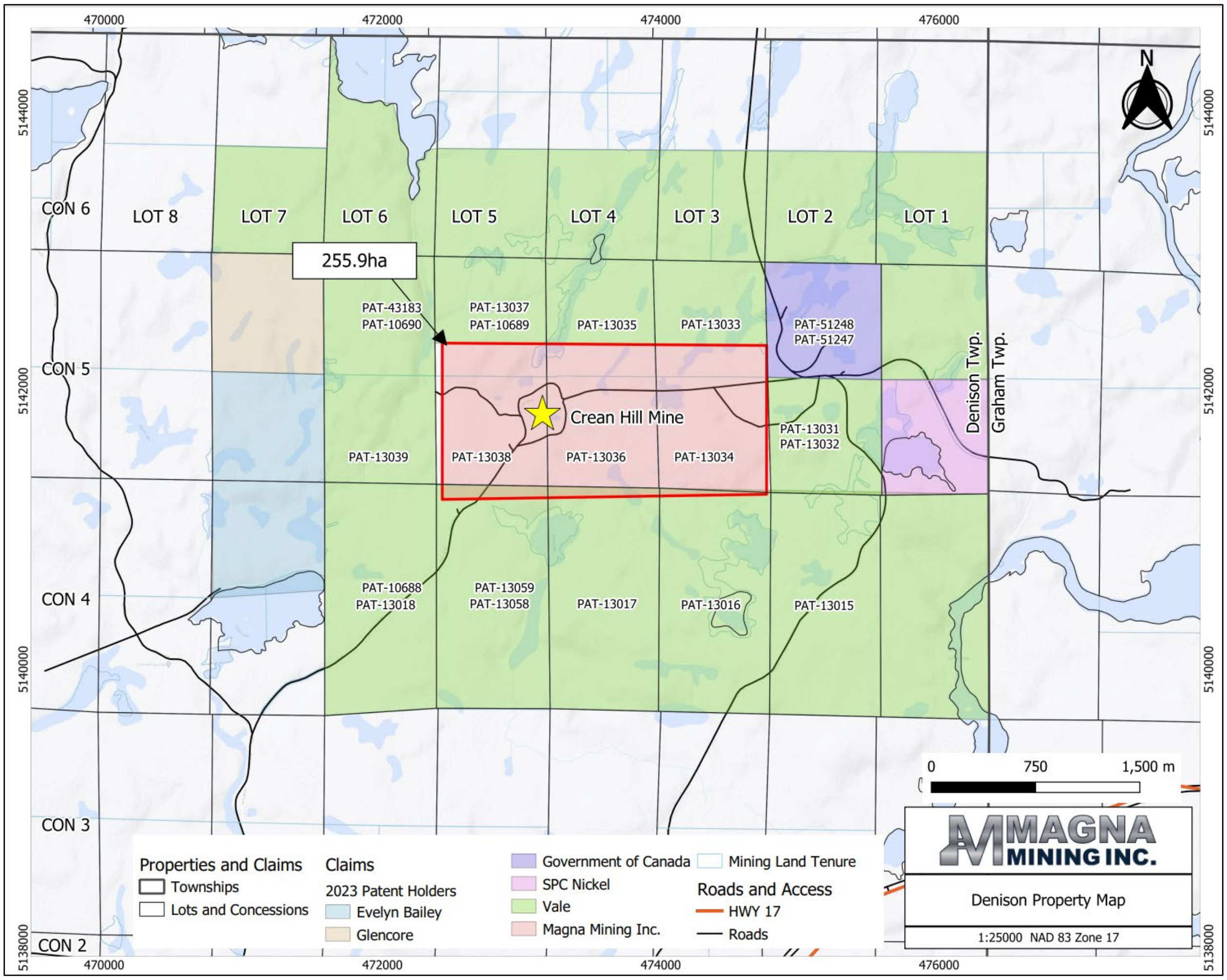
Figure 4-2: Property Location on Sudbury Area Map



4.2 MINERAL DISPOSITION AND TENURE RIGHTS

The Crean Hill Project property is an area of patented surface and mining rights, consisting of approximately 255.9 ha. It is located within the southern half of Lots 3, 4, and 5, and parts of the northern half of Lots 3, 4, and 5 of Concession 5, Denison Township, District of Sudbury. The area is parts 1 through 16 on registered plan 53R – 21031, filed with the Land Titles Division of Sudbury, as shown in Figure 4-3.

Figure 4-3: Crean Hill Land Tenure



Magna holds the mining rights from the top of the existing concrete capped Shaft #2 (former historic Crean Hill Mine production shaft) to a depth of 4,500 ft (1371.6 m). Vale Canada Limited (Vale) continues to hold all Mining Rights below 4,500 ft, from the top of the concrete capped Shaft #2.

The property is subject to surface easements as described in PIN No. 73382-0487(LT), PIN No. 73382-0537(LT) and PIN No. 73382-550(LT), and as represented on the survey plan 53R – 21031.

The property is legally described as follows:

- PIN No. 73382-0487(LT) being PCL 450 SEC SWS; NI / 2 LT 3 CON 5 Denison except L TI 6817; Greater Sudbury; subject to an easement as in SD202334.
- PIN No. 73382-0537(LT) being PCL 428 SEC SWS; NI / 2 LT 4-5 CON 5 Denison; SIT D422; Greater Sudbury.
- PIN No. 73382-550(L T) being LT 1-6 CON 4 Denison; S 1 / 2 LT 3-5 CON 5 Denison; SIT S48617, S62072, S63396, S89248; Greater Sudbury.

4.3 ONTARIO PROPERTY CLAIM STATUS

Magna has stated that the mining rights described previously in Section 4.2 are all in good standing.

4.4 UNDERLYING AGREEMENTS

The Crean Hill Project is wholly owned and controlled by Lonmin Canada Inc. (Loncan, a subsidiary of Magna) as of July 2018, when the joint venture between Lonmin (Loncan's predecessor) and Vale was cancelled. The joint venture was established in 2005 with the intent of exploring multiple Vale properties for low-sulfide, high-PGE-Au mineralization, as it was believed they hosted significant exploration potential. These properties included Capre, Denison (Crean Hill), Levack North, McKim, Trillabelle, and Wisner.

Vale reserves a three-percent Net Smelter Return royalty from the sale or other disposition of any metals or non-metallic minerals, or other materials mined, produced, or otherwise recovered from the Revised Property (or any waste rock or tailings derived from the Revised Property). This royalty is on, in accordance with, and subject to the terms set out in the Royalty Agreement.

From and after the completion of the Beneficial Transfer, Loncan will have the right to reasonable access to, egress from, and use of (such right to access and egress subject to certain terms and conditions set forth in this Agreement and the Ancillary Agreements) such parts of the Surface Rights and other adjoining surface rights of Vale. This may be reasonably required from time to time by Loncan, and reasonably agreed to by Vale Canada, to permit early exploration, advanced exploration, and mine operations to be conducted by Loncan or its agents in or on the Revised Property.

Vale reserves and has the right to access, upgrade (if required), operate, and use the Crean Hill Mine surface and underground infrastructure (for persons and vehicles, and with or without tools, equipment and machinery) in the event of a decision by Vale to conduct any early exploration, advanced exploration, or development or mine operations in the future on, in, or under the property or any other adjacent or proximate property of Vale Canada (including below the Crean Hill Cut-off Depth). This must be subject to and in accordance with a Crean Hill Mine access agreement, which must be negotiated in good faith and entered between Vale and Loncan at that time. The agreement must consider the relative existing and proposed operations and facilities of each Vale and Loncan on, in, under, or adjacent or proximate to the Revised Denison Property and the property, and such other matters as are reasonably relevant at that time.

Loncan must first offer Vale the right to process and / or purchase the ore or metals from ore mined by Loncan from the Revised Property before offering a contract on market terms with a third party to process and / or purchase ore.

The Author is not aware of any other underlying agreements relevant to the property.

4.5 MAGNA ACQUISITION OF LONCAN

On 16 August 2022, Magna announced it had entered into a definitive share purchase agreement (the Purchase Agreement) to acquire 100% of Loncan, including the Denison (Crean Hill) Project and the past producing Crean Hill Ni-Cu-PGE mine.

On 7 November 2022, Magna announced that it has closed the acquisition of Loncan, including the Denison Project and the past producing Crean Hill Ni-Cu-PGE mine. This was pursuant to a share purchase agreement dated 15 August 2022 between the Corporation, Loncan, and each of the shareholders of Loncan and Sibanye UK Limited, as shareholder representatives.

Under the terms of the Purchase Agreement, Magna acquired 100% of the issued and outstanding shares of Loncan—whose core asset is the Denison Project—in exchange for an aggregate purchase price of $16 million. The purchase price was comprised of a closing payment of $13 million in cash (the First Payment) and a deferred payment of $3 million (the Deferred Payment), payable pro rata to each shareholder of Loncan (the Vendors).

The Deferred Payment is payable on or before the 12-month anniversary of the closing of the acquisition. The Corporation will use commercially reasonable efforts to settle the Deferred Payment in cash, but may settle the Deferred Payment in common shares of the Corporation priced at the time of issue, in accordance with the rules of the TSX-V.

As ongoing security pending the settlement of the Deferred Payment, the Corporation has granted a pledge of the shares of Loncan in favour of the Vendors. The Corporation inherited Loncan's existing commercial arrangements with Vale, including access rights and certain net smelter return royalties. Other arrangements, including Loncan's joint venture arrangements with Wallbridge Mining Company Limited, terminated concurrently with the completion of the acquisition.

4.6 ONTARIO PERMITS AND AUTHORIZATION

The Ontario Mining Act regulations require exploration plans and permits, with graduated requirements for early exploration activities of low to moderate impact undertaken on mining claims, mining leases, and licences of occupation. Exploration plans and permits are not required on patented mining claims.

As the property is on patented land, exploration plan and permit applications under the Mining Act are not required by Ontario’s Ministry of Energy, Northern Development and Mines (MENDM), for exploration and advanced exploration work. The property is also considered an active mining area, where any mining activities that fit within the current closure plan may commence without additional permitting.

The Author is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the exploration work recommended for the property.

4.6.1 Exploration Plans and Permits Required under the Mining Act

The Ontario Mining Act regulations require exploration plans and permits, with graduated requirements for early exploration activities of low to moderate impact undertaken on mining claims, mining leases, and licences of occupation. Exploration plans and permits are not required on patented mining claims such as the Crean Hill Project.

There are several exploration activities that do not require a plan or permit, and that may be conducted while waiting for a plan or permit to become effective. These include the following:

- Prospecting activities such as grab / hand sampling, geochemical / soil sampling, and geological mapping.
- Stripping / pitting / trenching below thresholds for permits.
- Transient geophysical surveys such as radiometric or magnetic surveying.
- Other baseline data acquisition such as taking photos and measuring water quality.

4.7 PROPERTY ENVIRONMENTAL CONSIDERATIONS

The Project is a historical mine site with a filed closure plan from the MENDM. Approximately 20.4 Mt of ore was extracted from Crean Hill during its operating lifespan. Following closure in 2002, the site surface infrastructure was removed to prepare for site remediation, including the headframe, backfill plant and other buildings, fixed mining infrastructure, power lines, and rail lines. Since then, significant decommissioning work has been undertaken, including the following:

- Shafts, raises, and other openings to surface were capped with concrete.
- Waste rock was relocated to the Crean Hill Main Site Open Pit and Ellen No. 2 Pit.
- Crean Hill Main Site Open Pit was capped with clay, contoured, revegetated, and fenced.
- Disturbed areas were vegetated.
- The former mine landfill was capped, and a seepage barrier was installed.

The Author is not aware of any environmental liabilities related to the historic operation that are the responsibility of Magna.

As far as the Author is aware, the environmental liabilities related to the Project, if any, are negligible.

The Author is unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform exploration work recommended for the property.

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

5.1 ACCESSIBILITY

The region in which the property is located is serviced by Highway 17—a component of the Trans-Canada Highway Network—and the Sudbury Regional Airport, which has daily regional flights to Thunder Bay, Toronto, Timmins, and Ottawa.

The Crean Hill Project is located 7 km north of Highway 17, and approximately 28 km southwest of the City of Greater Sudbury, Ontario, Canada. It is within the southern half of Lot 5, Concession 5 of Denison Township.

Existing established all-season roads are available to transport materials, equipment, and personnel to and from the site. The site is easily accessible by road throughout the year via Regional Road 4; north off of Highway 17 to Crean Hill Road. Figure 5-1 indicates the location of the Crean Hill Project in the western portion of the City of Greater Sudbury, and Figure 5-2 shows a more detailed view of the site and nearby infrastructure.

Figure 5-1: Location of Crean Hill within the City of Greater Sudbury

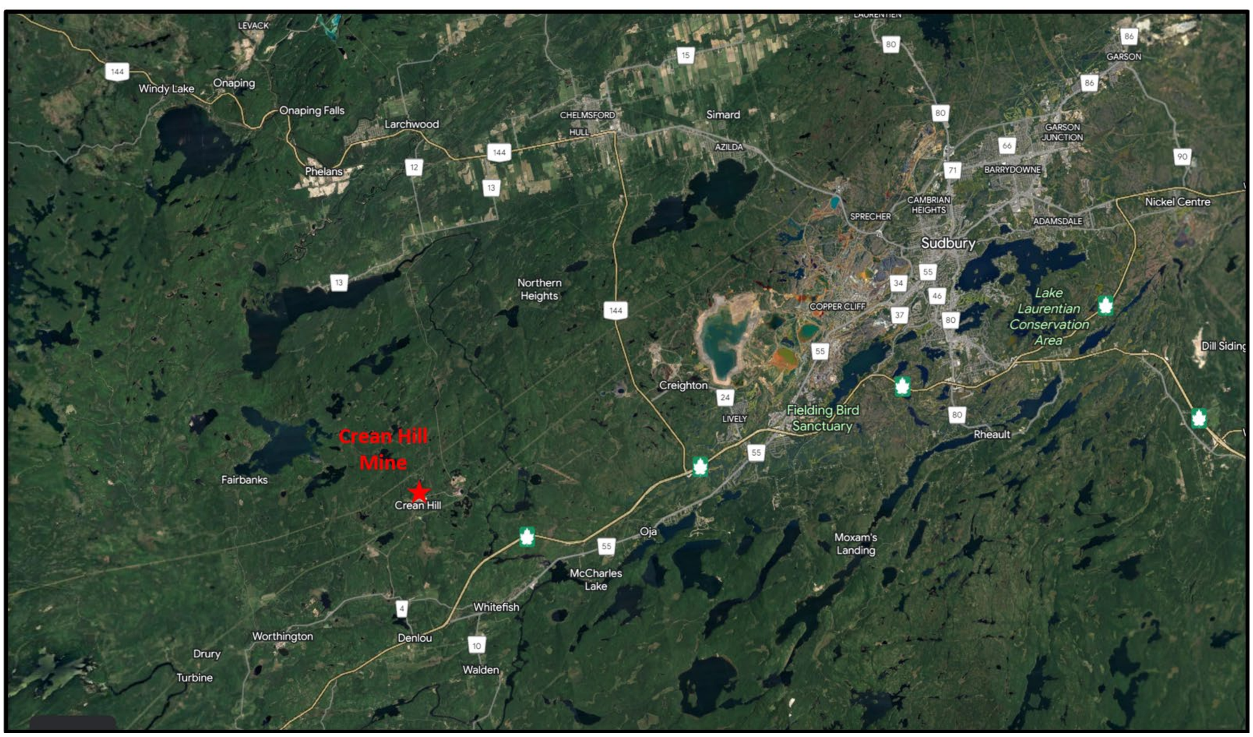
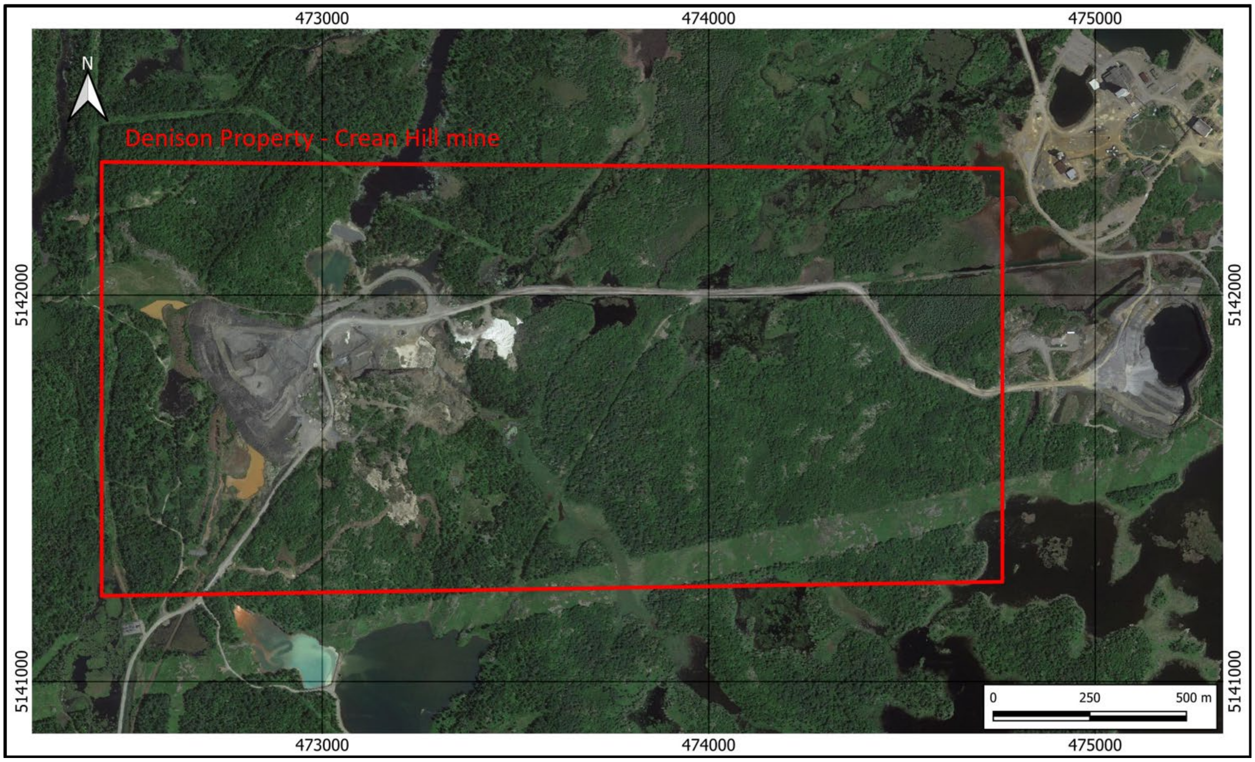


Figure 5-2: Site Map – Denison Property



5.2 LOCAL RESOURCES AND INFRASTRUCTURE

The City of Greater Sudbury is a major mining and manufacturing centre that has industry leading service providers, equipment suppliers and supply chains, and labour. The region has the infrastructure and technical resources to support any exploration and development work for the Project.

A 230-kV transmission line passes south of the property. A 115-kV transmission line passes at the western edge of the property with a substation at the property boundary.

Water is abundant in the region and can be sourced from nearby lakes and rivers to support exploration and mining activities.

There are areas within the property boundaries as well as other nearby sites to establish waste rock stockpiles.

5.3 CLIMATE

The active weather station closest to the project site is located at the Sudbury Airport, approximately 45 km northeast of the site. The climate in the region is typical Canadian Shield summers and winters, with daily average temperatures averaging from 19°C in the summer to -13°C in the winter. Precipitation comes in the form of 30 to 63 cm of snow per month in the winter (263 cm annual average), and 77 to 101 mm of rain per month in the summer (676 mm annual average) (http://en.climate-data.org).

Exploration and mining activities can be conducted year-round with appropriate measures to manage snow and cold weather during the winter months (generally November through March) and precipitation from April through October. Severe cold snaps (<-15deg.C) do not last long and typically last under 1 week. Winters are not considered severe and summer months provides excellent weather to enjoy multiple leisure activities and sports outdoors. The climate in Sudbury follows a definite winter-spring-summer-fall cycle.

5.4 PHYSIOGRAPHY

The property lies at a mean elevation of about 290 masl (metres above sea level). Relief is moderate and typical of Precambrian Shield topography.

6 HISTORY

The history on the property dates to 1885 when the mineralization was discovered by Francis Crean. The Project was subjected to sporadic exploration and production between 1906 and 2023 by various operators.

6.1 HISTORICAL EXPLORATION

Historical exploration activities are summarized in Table 6-1.

Table 6-1: Historical Exploration Activities completed at the Crean Hill Property

Year(s)	Company	Activity
1885	Francis Charles Crean	Discovered the Crean Hill deposit.
1906–1918	Canadian Copper Company	First production from the Crean Hill open pit and underground mining began at a rate of 300 tonnes / day. A total of 1.15 million tonnes at 2.07% Ni and 2.35% Cu was produced.
1918	Inco Limited	The property was transferred to the International Nickel Company (INCO).
1919	Inco Limited	Crean Hill Mine was closed.
1938	Inco Limited	Surface exploration drilling
1950	Inco Limited	The Crean Hill underground workings were dewatered, and underground diamond drilling commenced.
1954	Inco Limited	Airborne electromagnetic (EM) and magnetic surveys were carried out as part of the 1954 regional geophysical program.
1956–1957	Inco Limited	The Crean Hill No. 2 Shaft was collared and sunk to a depth of 2,116 ft (645 m).
1958	Inco Limited	Crean Hill development of No 2 Shaft was completed and the mine was subsequently closed.
1965- 1971	Inco Limited	Crean Hill development recommenced, and production reached a rate of 3,860 tonnes / day. No. 2 Shaft was extended to a depth of 4,180 ft (1,274 m). A total of 10.5 million tonnes at 1.05 %Ni, 0.89 %Cu, and 1.47 g/t PGE-Au was produced underground with an additional 1.1 million tonnes at 0.73% Ni and 0.56% Cu produced from the open pit.
1972 - 1978	Inco Limited	Crean Hill Mine was closed and re-opened again as development work continued until 1978 and then closed.
1983 - 1986	Inco Limited	Drilled 45 holes totaling 15,436 ft (4,705 m) in the immediate vicinity of the Vermilion Mine site. The program intersected erratically distributed Cu-Ni-PGE mineralization. Magnetometer and electromagnetic (EM) Very Low Frequency (VLF) surveys were completed over the property.
1984	Inco Limited	Surface mapping on 800 ft (244 m) spaced lines was completed on the Denison property, focusing on Cu and Ni.
1985	Inco Limited	Geophysical work included induced polarization (IP) and VLF surveys on select areas of interest outlined in the 1984 mapping.

Year(s)	Company	Activity
1986	Inco Limited	Surface diamond drilling was completed to test the Crean Hill Ni and Cu-PGE- Au targets.
1987	Inco Limited	Crean Hill Mine was reopened.
1989	Inco Limited	A shallow drill program was conducted in the footwall side of the deposit south of the Crean Hill Main orebody to test the potential for precious metals enrichment. No significant new zones of mineralization were encountered. Shallow drilling was also conducted in the Ellen environment immediately to the east of Crean Hill and up-dip from the Glencore Lockerby Mine.
1993	Inco Limited	Borehole EM surveys were completed in three Crean Hill underground drillholes testing the down-plunge continuity of the known Ni zone. Two exploration drillholes were completed to test the contact environment below the 5,000-ft level, between the Crean Hill and Lockerby mines. No significant mineralization was encountered at the contact or in the adjacent footwall rocks.
1997	Inco Limited	Nine existing surface Ni-Cu target drillholes were surveyed with UTEM-4.
1998	Inco Limited	Main Zone Ni-Cu grab samples from the Crean Hill 3,840–3,980-ft (1,170–1,213-m) levels were submitted for mineralogical analysis. Five additional samples, from each of the two composite mill test samples from the 2,550–2,800-ft (777–853-m) and the 3,840–3,980-ft (1,170–1,213-m) levels, were also analyzed.
1999	Inco Limited	Two Vermilion surface drillholes were surveyed using UTEM-4.
2001	Inco Limited	A resource estimate for the Crean Hill 9400 Zone was completed. A 9400 Zone exploration drilling program was started late in the year. One hole was completed with no significant intersections.
2002	Inco Limited	Between 1987 and 2002, a total of 7.62 million tonnes at 1.25 %Cu, 1.64 %Ni, and 2.14 g/t PGE-Au were produced from the Crean Hill Main, Intermediate, and West orebodies. A drill program consisting of 3,406 ft (1,038 m) of BQ core from ten underground 1,000-ft (305-m) level drillholes, and 7,260 ft (2,212 m) of NQ core from four surface holes, was completed to confirm and explore for extensions of the Crean Hill 9400 Zone. The mine was subsequently closed and decommissioned.
2003	Inco Limited	In 2003, the Lonmin-Vale Joint Venture was initiated, and their focus included the Denison property, specifically searching for platinum group metals (PGM). Property scale mapping and sampling were conducted to establish a detailed lithological and structural map. Surface UTEM and IP surveys were conducted. Five boreholes were surveyed with borehole UTEM-4 and 14 holes were surveyed with down-hole IP (0.125 Hz).
2005	Inco Limited	The 2005 drill program consisted of 18 holes totaling 18,720 ft (5,706 m), testing the depth extensions of the known Vermillion mineralization, the strike and plunge extensions of the Crean Hill 9400 Zone, and MIM Distributed Acquisition System (MIMDAS) IP chargeability anomalies in the near surface environment.
2006	Inco Limited	The mapping and sampling program was continued and included a focus on the Vermillion Mine area. A total of 165 grab samples was collected for geochemical and thin section analysis, yielding numerous anomalous PGE-Au occurrences, all of which are located within a corridor approximately 80 m south of the Sudbury Igneous Complex (SIC) contact. Most notably, these showings are centered around the west flank of the main Crean Hill embayment, the west flank of the eastern embayment, and the area

Year(s)	Company	Activity
		immediately south of the Beeper Zone. The 2006 drilling program, totaling 20,098 ft (6,126 m) was directed at investigating the strike and plunge extensions of the Crean Hill 9400 and 109 Zones, the depth extension of the Glencore Beeper Zone onto the Denison property, the up-plunge extension of the 8800 Zone, and the footwall potential of the Eastern Embayment.
2006	Vale Canada	CVRD of Brazil acquires Inco for an all-cash offering of $17 billion. Company rebranded as CVRD Inco.
2007	Vale Canada	Company rebranded as Vale Inco then Vale Canada. A limited amount of mapping was carried out around the Vermillion deposit with an emphasis on structure. Numerous down-hole borehole UTEM surveys were conducted on recently drilled holes within the 9400 and 8800 Zones. The 2007 drilling program, totaling 36,093 ft (11,001 m) was primarily directed at investigating the strike and plunge extensions of the Crean Hill 8800 and 9400 Zones. One borehole was completed and targeted the footwall potential of the Eastern Embayment. The understanding of the platinum-group elements (PGE) mineralizing systems at Denison was advanced.
2008	Vale Canada	A total of 19,705 ft (6,006 m) was drilled in 16 holes. The mineralized system was determined to extend from the 9400 Zone down-dip to the 99-Shaft Zone, but the tenor of mineralization, where tested, was determined to be sub-economic. The bottom of the 9400 Zone was also extended and better defined through additional drilling (9400 down-dip). The 101 Zone was tested along strike and down-dip. A new concept connecting the 101 Zone to the contact (101 Zone East Extension) was drill tested with positive results. A new concept was successfully drill tested in the footwall of the 109 Zone, resulting in the discovery of the 109 Footwall (FW) Zone. A total of 12 holes were UTEM surveyed in 2008, generating plates explained by known mineralization and mine workings. Optical televiewer survey on two boreholes in the 109 FW Zone confirmed orientation of mineralized features. The 9400 Zone Post Mining Designation (PMD) was updated to reflect the addition of the down-dip extension. The 8800 Zone Exploration Potential, last updated in 2006, was reduced in size to reflect the results of the 8800 Zone drilling conducted in 2007. A new zone, the 8800 contact Zone, discovered in the 2007 drilling.
2009	Vale Canada	The exploration program at the Denison property was primarily focused on follow-up to the late 2008 discovery of the 109 FW Zone. A total of 20,726 ft (6,317 m) was drilled in 29 holes. Drilling was directed toward defining the limits of the mineralized zone. A total of eight holes were surveyed by optical televiewer. There were no other geophysical surveys carried out at Denison in 2009. The 109 FW Zone was projected to surface, and the area was prospected. A 195 ft by 985 ft (60 m by 300 m) area was stripped, washed, mapped in detail, and channel sampled with numerous continuous Low Sulphide, High Precious Metal (LSHPM) results returned, confirming the continuity of the 109 FW Zone to surface.
2009	Vale Canada	Metallurgical test results, based on three 25-kg composite samples from a single hole, showed variable but favourable precious metal recoveries, generally in the high 70% to 80% range, assuming a standard flowsheet at Clarabelle Mill.
2010	Vale Canada	A total of 34,738 ft (10,588 m) was drilled in 58 drillholes, including the extension of two historical drillholes. Other advancements such as surface stripping, channel sampling, geophysical televiewer surveys (19 boreholes), geotechnical work, mineral resource modelling, and mineralogical and metallurgical studies were completed. The 109 FW mineral envelope was projected to surface, and the area was stripped, and channel sampled,

Year(s)	Company	Activity
		returning 32 samples >2.99 g/t total precious metals (TPM), and 8 samples >9.0 g/t TPM, with the highest grade sample assayed at 35.76 g/t TPM. In total, 291 channel samples were collected and assayed.
2011	Vale Canada	A total of 1,089 ft (332 m) was drilled in two boreholes. A conceptual target testing shallow potential PGM parallel to the 9400 Zone, called the 100 Zone, was tested. Lonmin fully vested in the joint venture in December 2011, earning a 50% interest in any declared LSHPM deposit on the Lonmin-Vale Joint Venture properties in the Sudbury Basin.
2012	Vale Canada	A total of 4,314 ft (1,315 m) was drilled, completing 12 boreholes, targeting the low-grade contact sulphide and potential LSHPM FW mineralization in the saddle zone, and geotechnical drilling in the highwall (HW) north of the existing Crean Hill pit. Drilling was suspended due to budget constraints. Acid rock drainage and pre-feasibility studies were completed by Klohn Crippen Berger and Tetra Tech Wardrop, respectively.
2013	Vale Canada	No work was completed on the Denison property; all studies were suspended.
2014	Loncan	Lonmin Canada Inc, a wholly owned subsidiary of Lonmin Plc, became the operator of the Vale-Lonmin Joint Venture, including the Denison property. A total of 30,610 ft (9,330 m) was drilled in 43 holes, with the primary goal of increasing confidence in the 109 FW Zone. Three holes targeted the saddle zone between the 109 and 101 Zones. Geotechnical data and specific gravity (SG) data were collected from most boreholes. The previously saw-toothed shape of the mineral envelope along the plunge of the hinge (southern margin) was remodeled and smoothed out with the intersection of significant mineralization in previously existing gaps.
2015	Loncan	A total of 46,257 ft (14,099 m) was drilled in 34 holes as part of drill programs aimed at the 109 FW Zone and 9400 Zone. Drilling in both zones aimed to increase confidence by targeting areas of low drilling density. In the 109 FW Zone, boreholes with significant assay results were wedged to duplicate and triplicate the intersection at short distances to provide short-range grade variability data and to provide material for geometallurgical testing. Geotechnical data was collected from most boreholes and specific gravity data was collected from all boreholes.
2016	Loncan	A total of 23,261 ft (7,090 m) was drilled in 63 holes in drill programs at Denison. Drilling in the 109 FW Zone concentrated on collection of larger diameter core for geometallurgical testing. Thirteen boreholes targeted the 9400 Zone in areas of lower drilling density, and boreholes with significant assay results were wedged to duplicate and triplicate the intersection at a short distance to provide short-range grade variability data, and to provide material for geometallurgical testing. Geotechnical data was collected from most boreholes and specific gravity data was collected from all boreholes. The morphology of the 9400-area mineralization was re-interpreted as a tabular body that branches at the western margin, with highest TPM grades over largest widths seen at the intersection of the branches.
2017	Loncan	A total of 18,586 ft (5,665 m) was drilled in 16 boreholes, targeting the 9400 Zone and extensions of the 9400 Zone up-plunge, immediately west of the Crean Hill West Orebody, which was largely mined out. Drilling was subsequently curtailed due to budget constraints. Geotechnical data was collected from most boreholes and specific gravity data was collected from all boreholes. Both the 109 FW Zone and 9400 Zone were subject to

Year(s)	Company	Activity
		mineralogical study by Cabri Consulting Inc., and a metallurgical study was completed on the 109 FW Zone by Blue Coast Research.
2018	Loncan	Vale Canada and Loncan agree to terminate the 2003 Joint Venture Agreement and sign the Denison Property Transfer and Development Agreement, whereby Vale Canada transferred 100% ownership of the Revised Denison Property to Loncan.
2019	Loncan	Loncan and Wallbridge Mining signed a definitive letter agreement whereby Lonmin Limited (a wholly owned subsidiary of Sibanye-Stillwater) appointed Wallbridge as operator of Loncan's advanced-stage Denison Property.
2022	Loncan	In November 2022, Loncan was acquired by Magna Mining Inc. as a wholly owned subsidiary.
2022	Loncan	Loncan completed 2000 metres of diamond drilling in 10 drillholes. Drilling was focused on the Intermediate Contact Zone, the 101 FW Zone, and the 109 FW Zone.
2023	Loncan	Loncan continues exploration drilling, metallurgical testwork, and completes Preliminary Economic Assessment.

6.2 SUMMARY OF HISTORICAL PRODUCTION

The Crean Hill Mine operated from 1906 to 2002 primarily under the ownership of INCO; historical production during this period is summarized in Table 6-2.

Table 6-2: Summary of Historical Production from the Crean Hill Mine, 1906 to 2002

Period	Type	Tonnes (M)	Ni %	Cu %	PGE-Au (g/t)
1906–1918	Open Pit and Underground	1.15	2.07	2.35	
1965–1971	Underground	10.5	1.05	0.89	1.47
1965–1971	Open Pit	1.10	0.73	0.56	
1987–2002	Underground	7.62	1.64	1.25	2.14
Total	**–**	**20.37**	**1.31**	**1.09**	**1.56**

Production was from the Main, Intermediate, and West (9400) zones, and focused on the Sudbury Igneous Complex (SIC) contact nickel-copper mineralization.

6.3 HISTORICAL MINERAL RESOURCE ESTIMATE

Prior to the 2022 Mineral Resource Estimate completed by SGS, there have been no publicly disclosed NI 43-101 mineral resource estimates (MRE) or technical reports published on the property.

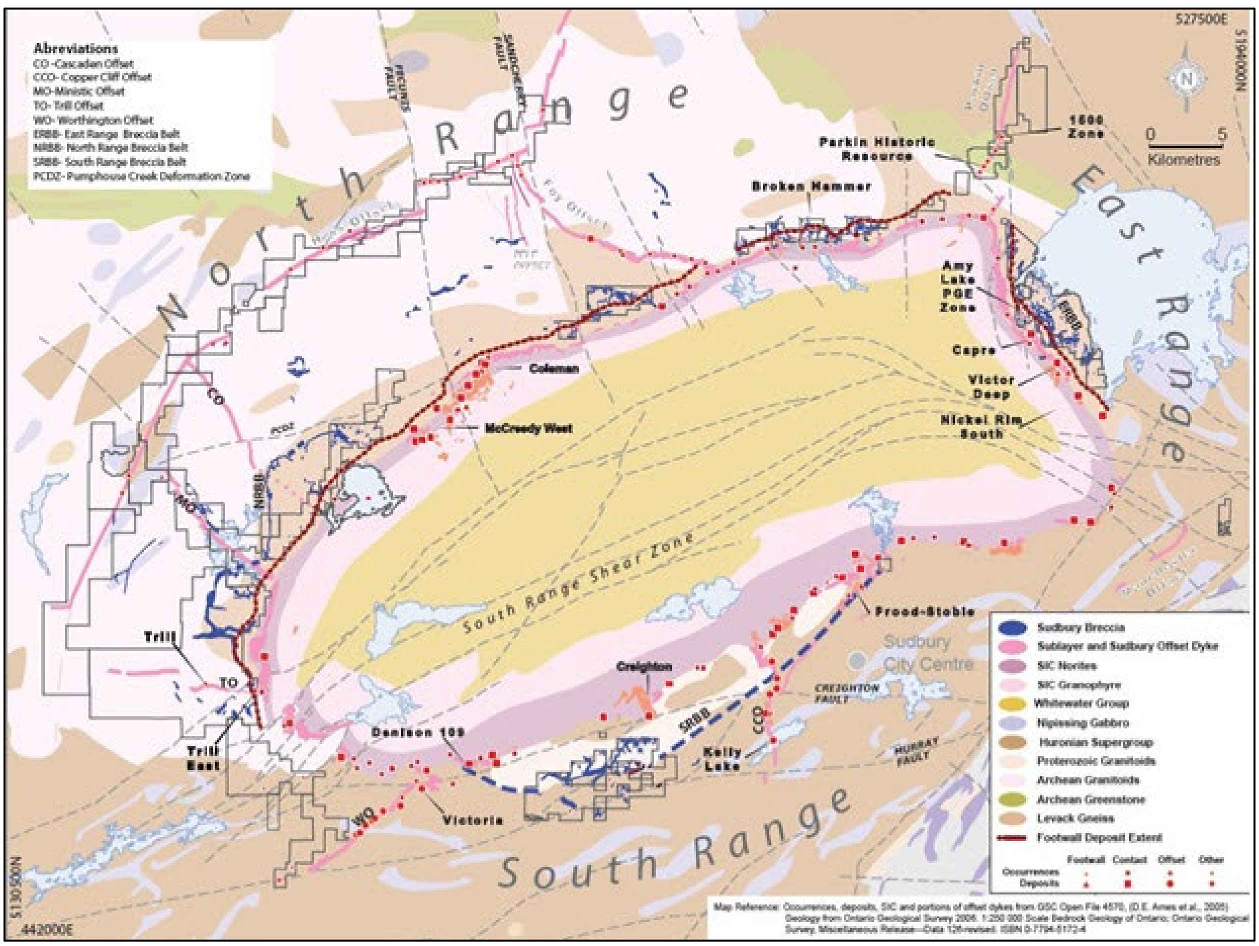
7 GEOLOGIC SETTING AND MINERALIZATION

The description of the geological setting and mineralization of the property in this section is sourced from SRK (2020), WSP (2020), and any other references therein.

7.1 REGIONAL STRATIGRAPHY

The accepted theory derived from volumes of data and study for the genesis of Ni-Cu-PGE deposits in Sudbury occur within the Sudbury Structure formed as a result of a major Early Proterozoic meteorite impact 1,850 million years ago (Ames and Farrow, 2007). The Sudbury Structure straddles the unconformity between Archean gneisses and plutons of the Superior Province and overlying Paleoproterozoic Huronian supra-crustal rocks of the Southern Province. It is geographically divided into the North, South, and East Ranges, as shown in Figure 7-1.

Figure 7-1: Simplified Regional Geology (Ames and Farrow, 2007)



The Sudbury Structure comprises four geologic domains:

- The Sudbury Igneous Complex (SIC) occurs as a 60 km by 27 km elliptical bowl-shaped body that formed from a meteorite impact melt sheet. It consists of a basal xenolithic norite breccia (contact sublayer) overlain by norite, quartz-gabbro, and granophyre. It has historically been referred to as the Nickel-Bearing Irruptive, the Sudbury Nickel Irruptive, and the Nickel Irruptive.
- Concentric and radial dykes of diorite, granodiorite, and quartz diorite.
- The FW to the SIC contains a zone, up to 80 km wide, of Archean and Proterozoic rocks that are fractured, brecciated (i.e., Sudbury breccia), and locally partially melted (e.g., Late Granite Breccia) or recrystallized due to the meteorite impact and subsequent emplacement of the SIC.
- The SIC is overlain by the Whitewater Group, comprising of fall-back impact debris forming super-crustal breccia of the Onaping Formation and the overlying basin-fill sedimentary rocks of the Onwatin and Chelmsford Formations.

The Crean Hill property is in the South Range of the SIC. The main mass of the South Range SIC consists of a lower unit of the quartz-rich norite. Stratigraphically above is the green norite with irregular bodies of brown norite, followed by the quartz gabbro and then the granophrye layers (Lightfoot, 2016).

A magmatic breccia—called the sublayer—is found at the basal contact of the main mass in embayment and trough structures.

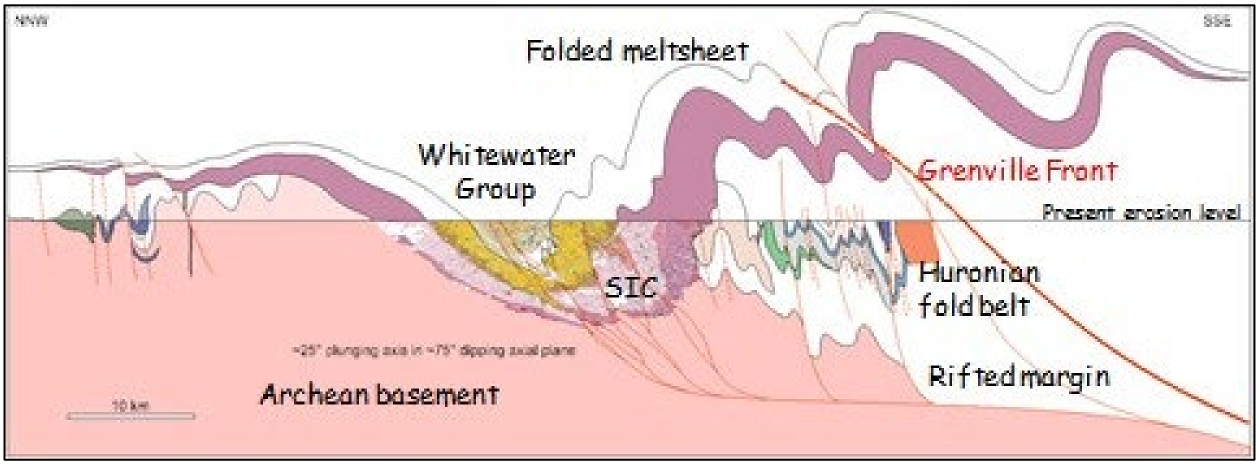
The footwall (FW) to the SIC South Range is the Southern Province. The geology can roughly be divided into the Early Proterozoic (~2,450 Ma) Murray and Creighton Granite Plutons and Huronian Supergroup (2,250 to 2,460 Ma) mafic and felsic volcanic and sedimentary rocks. In ascending stratigraphic order, the rock formations present are:

- Elsie Mountain (mafic volcanic and some interflow sedimentary rocks)
- Stobie (mafic volcanic and sedimentary rocks)
- Copper Cliff (felsic volcanic rocks)
- McKim (argillitic and arenaceous rocks)
- Ramsey Lake (arenaceous and conglomeratic rocks)
- Pecors (argillitic and arenaceous rocks)
- Mississagi (sub-arkose and arkosic sedimentary rocks)

The Creighton and Murray Plutons are intrusive into older Huronian volcanic and sedimentary rocks, mostly of the Elsie Mountain and Stobie Formations.

The South Range of the SIC and adjacent Huronian rocks, for the most part, dip vertically or steeply north or south. Stratigraphic tops generally face south away from the SIC and toward the Grenville Front. The South Range Shear zone and Creighton and Murray faults are the manifestation of the deformation events that have shaped the present-day South Range (refer to Figure 7-1 and Figure 7-2) (Bleeker et al. 2014).

Figure 7-2: Cross-Section Illustration of the Conceptual Deformation of the SIC (looking east) (Bleeker et al., 2014)



The age of the deformation, which has resulted in the current sub-vertical orientation of the Huronian rocks, has not been definitively established. The metasedimentary rocks are interbedded sparingly with mafic volcanic flows of the Elsie Mountain Formation and commonly with volcanic rocks of the Stobie Formation. Many of these interflow metasedimentary rocks are sulphide bearing. The sulphides are dominantly pyrrhotite, with minor amounts of pyrite and trace chalcopyrite.

South Range FW rocks are cut by several small diabase and gabbroic intrusions that are often difficult to distinguish in the field. These include Matachewan dykes, Nipissing intrusions, quartz diabase (trap dykes), and Olivine Diabase. Both the quartz diabase and olivine diabase dykes are younger than the SIC. The Archean and early Proterozoic basement rocks are all crosscut by Sudbury Breccia.

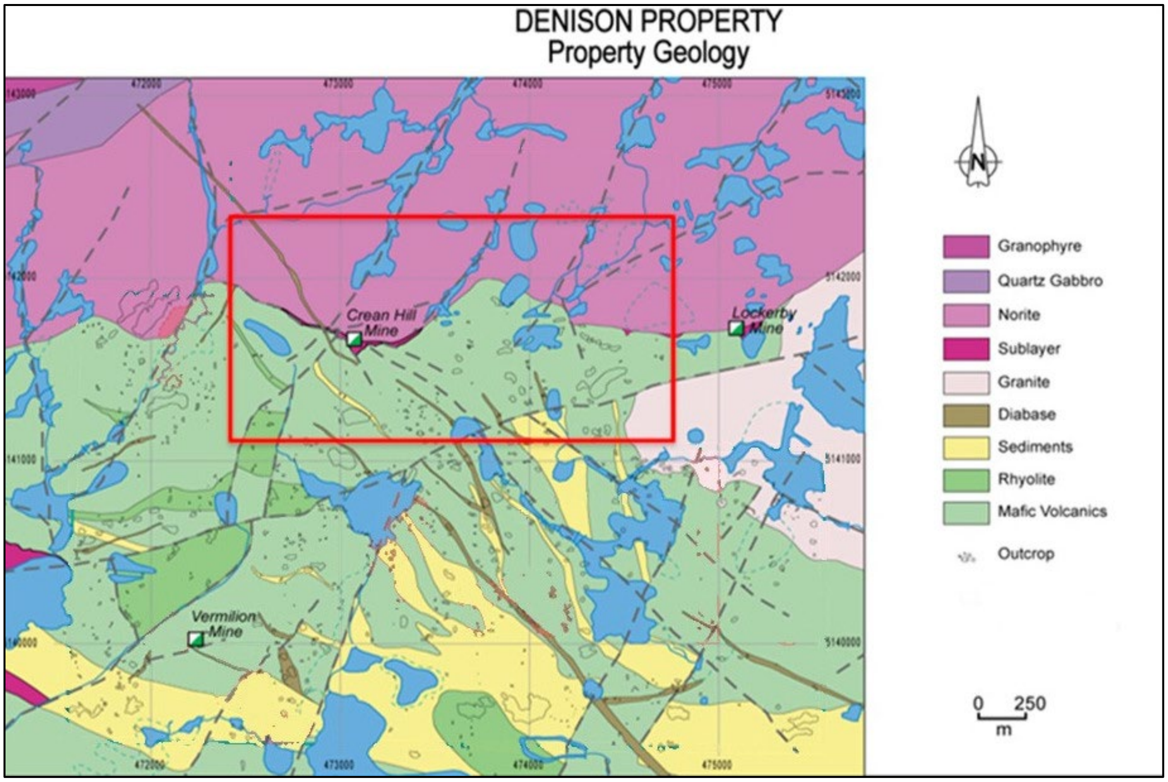
7.2 PROPERTY GEOLOGY

The property straddles the South Range of the SIC, approximately 30 km southwest of Sudbury, in Denison Township. From 1906-2002 a total of 20,370,000 tonnes of ore grading 1.09% Cu, 1.31% Ni, 1.56 g/t TPM was produced from the Main, Intermediate and 9400 zones.

The property hosts part of a large trough structure at the base of SIC, which contains a number of previously mined ore deposits including Crean Hill Main Orebody, Crean Hill Intermediate Orebody, Crean Hill West Orebody, Ellen Mine, and Lockerby Mine, each sitting in embayments (terraces) within the larger trough. The embayments largely control the distribution of Ni-Cu mineralization.

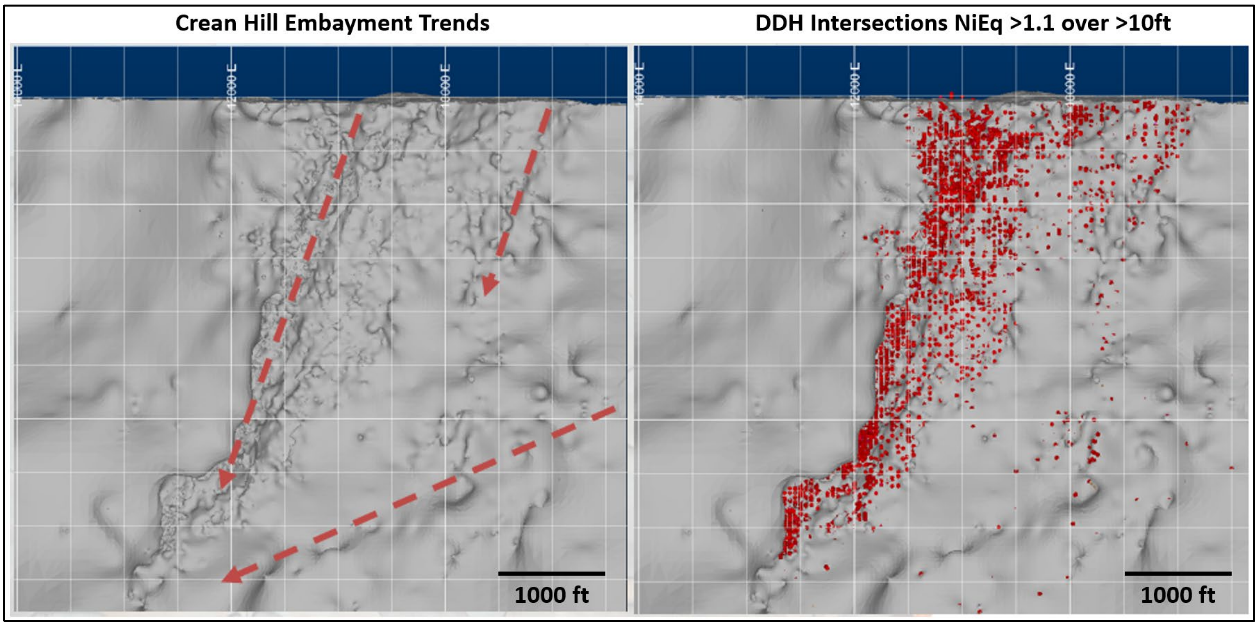
Additional embayments in the SIC containing significant Ni-Cu sulphide mineralization may be present at Crean Hill, in different orientations to the Mine Zone embayment (refer to Figure 7-3).

Figure 7-3: Denison Property Geology (Modified from Baker et al., 2015)



In the Creighton deposit (13 km along strike east of Crean Hill), at least three orientations of embayments are present: the steep plunging 400 embayment, the moderate east plunging 402 (Gertrude West) embayment, and the moderate west plunging 403 embayment.

Figure 7-4: Crean Hill Long Section (looking south) with SIC Contact in Grey and Arrows Displaying the Embayment Trends at Denison



The strike of the SIC contact ranges from 120° at surface to 80°, and the dip varies from steeply dipping to the north at surface through vertical, to steeply dipping over-turned to south at the lower depths. The contact between the SIC and the FW is very often sheared. Shearing and brittle faulting also occur within the FW, as well as local significant alteration (Baker et. al. 2015 and 2017).

A significant portion of the mineralization, such as the 109 FW Zones, the 101 Zone, and part of the 9400 Zone, are hosted in the FW rocks. The host rocks are dominated by metamorphosed basalt (historically mapped and logged as greenschist), but also include gabbro, andesite, rhyolite, and sedimentary units (arkosic quartzite and meta-pelite) of the Huronian Supergroup, Elsie Mountain Formation (Card et al., 1977). Minor lithologies include olivine diabase, quartz diabase (trap dykes), granite, schist, amphibolite, and Sudbury Breccia in the FW, and quartzose norite at the SIC contact.

Though the distribution of much of the mineralization in controlled by embayments, additional structural settings and controls may be present. The association between shear zones and Ni-Cu sulphide orebodies is common in the South Range of the Sudbury basin, with Ni-Cu sulphide orebodies in the Creighton and Garson deposits associated with large shear zones. The splays of the Crean Hill (Victoria) shear zone can be traced from through the 9400 orebody and into the Crean Hill Main open pit and appear to be associated with Ni-Cu sulphide mineralization at each.

At Crean Hill, the intersection between the Crean Hill shear zone and SIC is sub-parallel to the trend and plunge of the Crean Hill embayment, suggesting the Crean Hill shear zone may have controlled the formation of the embayment (SRK, 2006). The shear zones are associated with zones of alteration. In the western upper quadrant of the 9400 Zone mineral envelope, a wide zone of significant talc alteration is observed, affecting all rock types except olivine diabase and quartz diabase (trap dyke) (Baker, 2017).

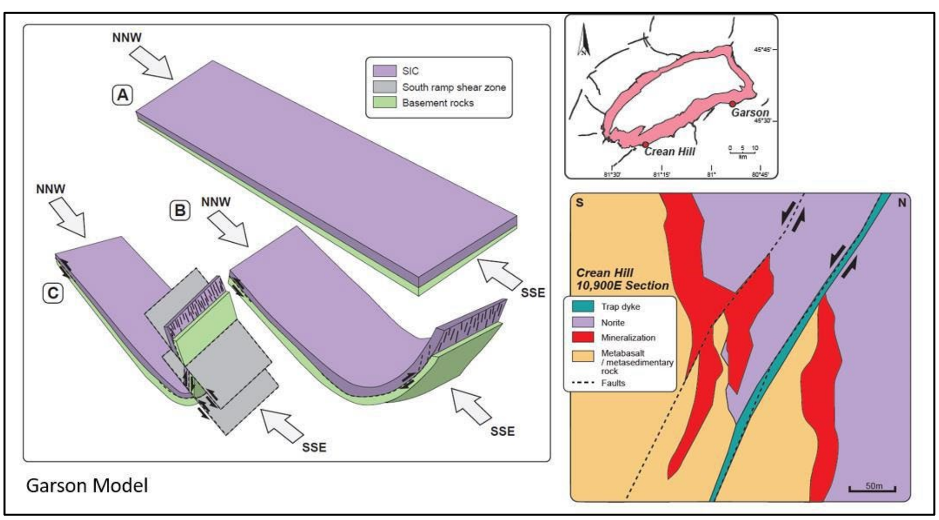
The Crean Hill shear zone may also control the distribution of PGE mineralization away from the Crean Hill embayment. In the South Range, for example, the Crean Hill, Creighton, Garson, Falconbridge, and Thayer Lindsey deposits all display shear zone controls on Ni-Cu sulphide mineralization. If the distribution of Ni-Cu sulphide mineralization is controlled by shear zones, it can be expected that the distribution of PGE mineralization may also be controlled by the shear zones. The PGE mineralization may be distributed within the shear zones along strike from the Ni-Cu sulphide mineralization, rather than directly into the deposit's FW. This is observed in the Garson deposit, with high tenor PGE mineralization observed in shear zones in the Garson ramp area, along strike from the main shear zone hosted Ni-Cu sulphide ores.

Two variably developed shear structures have also been observed along the limbs of the 109 FW Zone and are interpreted to form the pathway for mineralization of the FW; not as discrete mineralized features but rather as a route into the FW for migrating metals. The shears locally appear as chlorite and talc altered zones of metabasalt with strong foliation. The level of alteration is variable, with the extreme end member being very soft, heavily talc-altered beige intervals up to 30 cm, which have been encountered twice in drilling. Many intervals through the interpreted shears appear unaffected, with only typical levels of quartz-carbonate veining and alteration characteristic of the FW rocks (Baker, 2015).

There is one main fault in the immediate area of the 109 FW: a shallow fault striking 100° and dipping 25° south. This fault is comprised of two or more anastomosing horizons, where core is broken up along poorly healed joints, with local chlorite rich gouge horizons, bleached core, and locally significant quartz-carbonate veining. Locally, there are void spaces within the fault which are reported to have caused the abandonment of one hole in a previous drilling campaign. There is no offset of the 109 HW or 109 FW zones through this fault horizon; it appears to be a zone of weakness and alteration with no apparent offset. There is also no apparent trend in terms of enrichment or depletion of the 109 FW Zone mineralization due to the fault (Baker, 2015).

Sectional interpretations at Crean Hill are consistent with imbricate reverse fault slices stacked north over south. Figure 7-5 shows the evolution of the SIC at Garson.

Figure 7-5: Evolution of the SIC at Garson (Lightfoot, 2016)



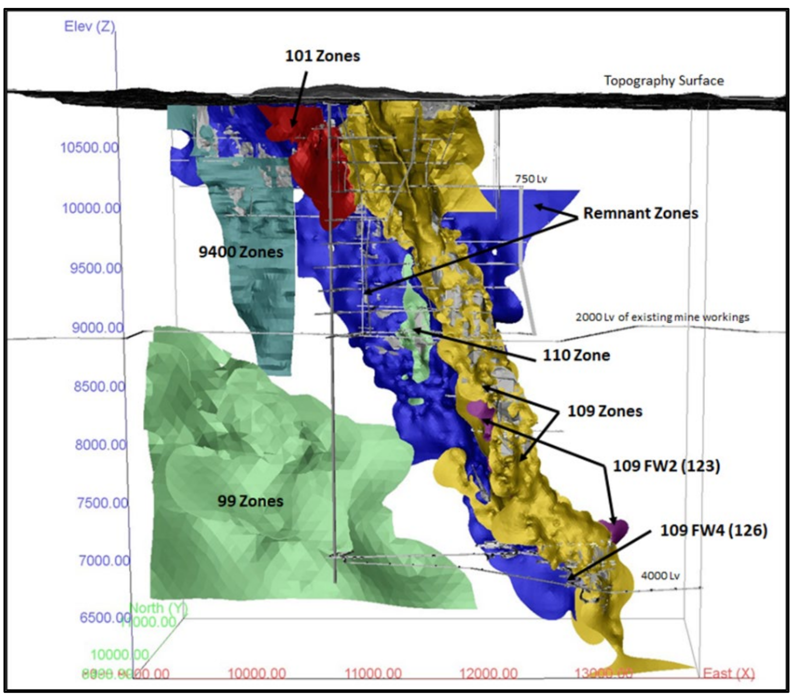
A similar, but not identical, model may apply at Crean Hill (Lightfoot, 2016). Many structures have a west-east trend and run close to the base of the SIC (e.g., the Victoria Shear which appears to have a dextral reverse motion), and there may also be splays of the Cliff Lake Fault (which typically exhibits south over north thrusting through much of the South Range, except where the basal contact is very steep). It is unclear to what extent the structures deformed / displaced the mineralization versus provided a pathway for the mineralization to follow (Lightfoot, 2017).

7.3 MINERALIZATION

The main mineralized zones from east to west (refer to Figure 7-6) are as follows:

- 109 W / Remnant Zones
- 126
- 123
- 109 FW
- 109 HW
- 99 Zone
- 101
- 9400
- 9400 FW Ext

Figure 7-6: Isometric View Looking North: Main Denison Deposit Models (grid is in ft.)



7.3.1 9400 Zone

The 9400 Zone mineral envelope, as currently defined, is 1,970 ft (600 m) in depth extent, up to 820 ft (250 m) in strike length, and ranges from 10 ft to 130 ft (3 m to 39 m) thick. The envelope extends from 10,470 ft elevation down to 8,500 ft elevation, or from 450 ft to 2,460 ft (150 to 750 m) below surface.

The zone occurs primarily down-dip of the historic Crean Hill West Orebody, as well as mineralization to the west of the mined stopes. It is a tabular body that curves to the south at depth, and thickens from east to west, branching into two to three apophyses at the western margin of the zone. The Ni-Cu rich and PGM poor eastern part of the 9400 Zone is in contact with the SIC, trending obliquely away from the contact into the FW to the west. Mineralization at the eastern part consists mostly of semi-massive to massive Contact-style pyrrhotite, pentlandite, and chalcopyrite.

Toward the west, into the FW, the PM grades increase whereas the Ni and Cu grades decrease. Here, the sulphide mineralization occurs as stringers, fracture-controlled within quartz / carbonate veins, disseminations within the host rock, and disseminations within quartz / amphibole veins / patches that are interpreted to be partial melts. The majority of the 9400 Zone is composed of this type of FW mineralization. FW sulphide mineralization is dominated by chalcopyrite and pyrrhotite. Other minor sulphide/arsenide minerals include pentlandite, pyrite, gersdorffite, and trace minerals identified primarily in thin section sphalerite, galena, bornite, chalcocite, cobaltite, sperrylite, michenerite, and merenskyite (Baker et al., 2017). This style of mineralization changes to the west, gradually becoming lower in sulphide and shifting to higher Pt, Pd, and Au grades.

While there is visual evidence of possible hydrothermal processes—including intimate spatial association of PM with alteration minerals seen in thin sections and the presence of structures that would allow for the movement of fluids and local pervasive alteration—there is no indication of a spatial fractionation process within the geochemical dataset, as might be expected if a secondary process took place. Rather, the low- and high-sulphide mineralization styles were likely emplaced into their current relative locations during a single mineralizing event (Lightfoot, 2017). The 9400 FW Ext Zone is interpreted to be a continuation of the 9400 Zone.

The FW rocks have been metamorphosed to greenschist and amphibolite facies. Locally, volcanic sections contain patches and veins of medium- to coarse-grained amphibole in a fine-grained quartz matrix. These are interpreted to be partial melts of FW lithologies due to contact metamorphism from the cooling SIC (Dressler, 1984). Common alteration styles include pervasive chlorite alteration of volcanic rocks, pervasive silicification of sedimentary rocks and rhyolites, and calcite +/- quartz +/- chlorite veins in the volcanic and sedimentary rocks. Less common alterations include pervasive talc and sericite alteration of sedimentary and volcanic rocks (Baker et al., 2017).

The low-sulphide Pt-Pd-Au style of mineralization was not well sampled in drilling prior to 2003. As a result, the upper half of 9400 Zone remains open to the west and to surface. The 9400 could also be considered open at depth as one possible interpretation of the 9400 Zone mineralized trend is that it extends into the 99 Zone at depth.

7.3.2 109 FW Zone

The Crean Hill 109 FW zone rests in the immediate FW of the main embayment structure, which hosted what was the Crean Hill Mine Main zone but now includes the 109 HW zone. There is little or no separation between the norite-hosted, semi-massive to massive mineralization of the 109 HW, and the much lower sulphide Pt-Pd-Au mineralization in the FW, hosted primarily in metabasalt (Figure 7-6).

The mineralization is often associated with partial-melt veinlets, thought to be a thermal effect from the emplacement of the SIC, and occurs with veinlets of chalcopyrite-pyrrhotite and local pentlandite near the SIC contact. Fine to 1-cm-thick quartz-carbonate veinlets are found throughout the deposit, often hosting pyrrhotite and local chalcopyrite. Fine disseminations and veinlets of gersdorffite are found locally.

Where present, metasediments and felsic metavolcanics are not as prospective as the metabasalt unit. The deposit has the morphology of an open fold with thin limbs and a thickened axial hinge in the FW of the apex of the Crean Hill embayment. However, the deposit is understood to be located by two shears running parallel to the limbs, concentrating a PGE-Au mineralization halo around the contact mineralization. The mineralization is particularly concentrated in the hinge at the intersection of the two limb shears.

Mineralization can be extremely low in sulphide toward the FW margin of its known envelope. The mineralization is not necessarily hosted in noticeably sheared rock, but rather the sheared areas define the mineralized corridors. The shearing may have prepared the host rock to receive—or acted as a conduit for—mineralizing fluids (Baker et al., 2015).

Because the low-sulphide Pt-Pd-Au style of mineralization was not well sampled in drilling prior to 2003, the 109 FW zone one remains open below the current extent to depth, along within the FW to the Main zone.

7.3.3 Remnant Zones

The Remnant, 109W / Remnant, Main Remnant, and the 109 HW Zones are the unmined Ni-Cu rich contact sulphide mineralization of the historic West, Intermediate, and Main orebodies. They are concentrated at or near the base of the SIC or within embayment structures, and associated with sublayer norite and quartz-rich norite phase of the SIC. Generally, these zones become more Cu rich the further the mineralization is from the main contact mineralization.

The main sulphide assemblages are massive to net-textured pyrrhotite, pentlandite, and chalcopyrite mineralization, where pentlandite is the main Ni-bearing mineral, and chalcopyrite is the main Cu-bearing mineral. Most of the Remnant Zone is found outside of the main embayment structures and therefore is generally lower grade relative to the 109W / Remnant and Main Remnant, which are generally within or proximal to the main embayment structure.

A portion of the current 109W / Remnant Zone that extends west of the 109 FW was previously modelled by Vale as a separate zone they called the 109 W; however, the current interpretation is that this and the Remnants are part of a continuous zone. The Remnant, 109W / Remnant, Main Remnant, and the 109 HW Zones are oriented sub-parallel to one another and the main embayment trend. It is unclear whether these zones represent imbricate reverse fault slices stacked north over south, whether these zones formed as a result of the mineralization exploiting pre-existing structures, or a combination of both.

Because the low-sulphide Pt-Pd-Au style of mineralization was not well sampled in drilling prior to 2003, large portions of the FW to the Remnant Zones remain untested for this style of mineralization.

7.3.4 123 (109 FW2) and 126 (109 FW4) FW Zones

The 123 and 126 Zones (Figure 7-6) are narrow mineralized zones starting at approximately 800 m depth, oriented sub-parallel and between the Main Remnant Zone and the Remnant Zone. These zones are likely formed by similar mechanisms and have similar mineralogy as those adjacent zones.

7.3.5 101 Zones

The 101 Zones have been modeled as four parallel mineralized structures extending out from the SIC contact from near surface to approximately 400 m depth and up to 200 m southwest, perhaps exploiting weakness along lithological contact in the footwall rocks. The orientation and metal ratios are curious for this zone. Unlike most of the mineralized zones which are near parallel to the SIC contact or following identified embayment trends, the strike orientation of the 101 Zones is oblique to the main mineralized trend. Also peculiar is the high Ni/Cu of the zone despite extending so far into the footwall. The 101 Zones remain partially open along to the southwest to depth.

7.3.6 99 FW Zone

The 99 FW Zone has been modeled as three sub-parallel mineralized zones, are oriented sub-parallel to the SIC contact, and have a strike extent of over 1,800 m and a depth extent of over 700 m within the Property boundary. The largest and most continuous of the three zones is located at the SIC basal contact and two smaller zones are interpreted to be within the footwall. The thickest part of these zones has been interpreted to be plunging shallowly to southeast along a secondary embayment structure. There is a lower confidence in the interpretation of the 99 FW Zone because of the limited number of drillhole intercepts and the high angle at which the drilling completed was oriented relative to the zones.

8 DEPOSIT TYPES

The description of the deposit model for the property in this section is sourced from SRK (2020) and any other references therein.

Historical production over the past 125 years, plus current reserves in the Sudbury mining districts, have been estimated at approximately 1.6 billion tonnes of ore, containing over 60 million ounces of PGM plus Au, over 11 million tonnes of Ni, and over 10.8 million tonnes of Cu (Lightfoot and Farrow, 2002; Eckstrand and Hulbert, 2007; Ames and Farrow, 2007; Lightfoot, 2016).

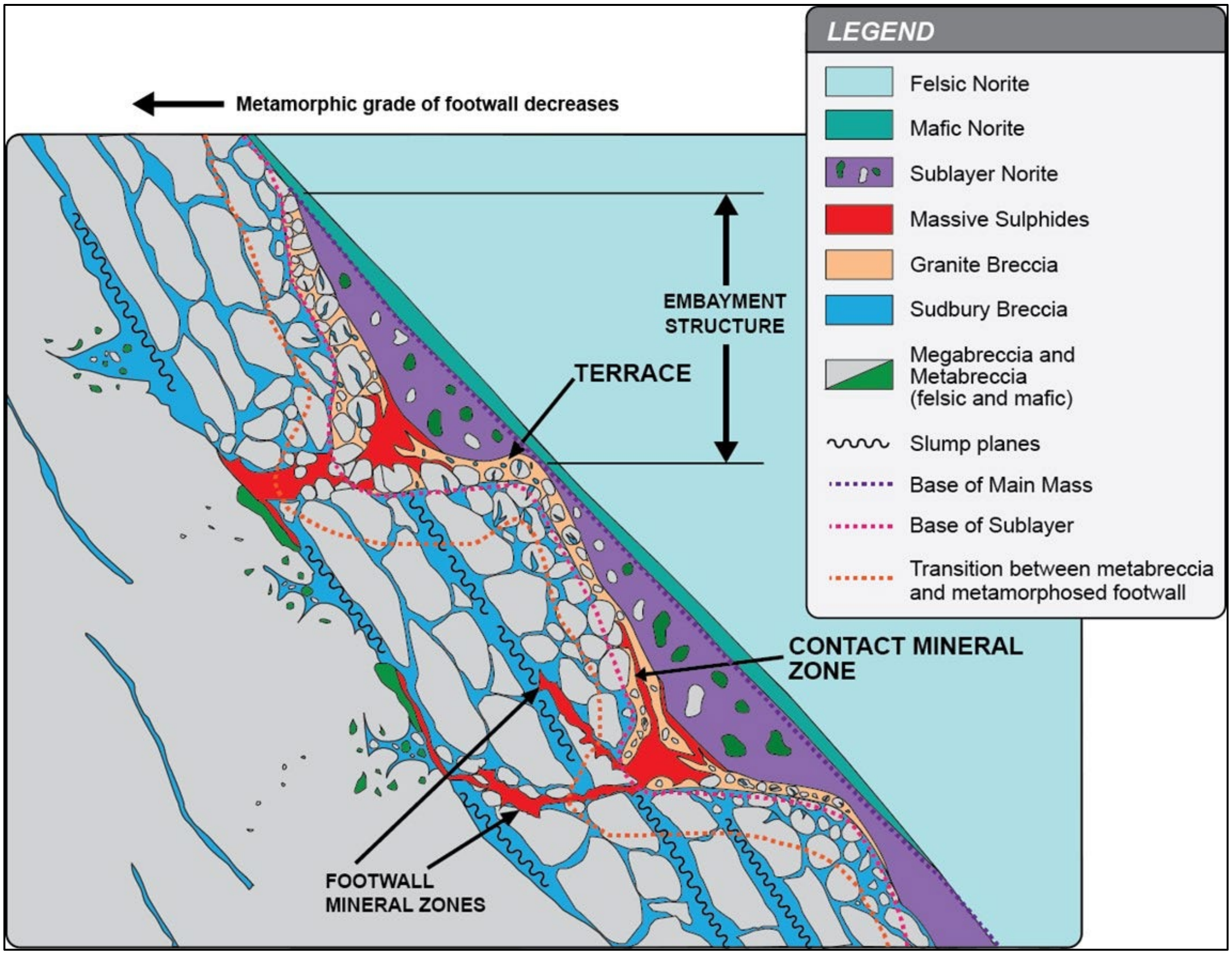
There are several main types of mineral deposits in the Sudbury area:

- Contact deposits, including massive sulphide consisting of Ni, Cu, Co, Pt, Pd, and Au mineralization along the lower contact of the SIC, both within the contact sublayer and in the immediately adjacent FW Breccia.
- FW deposits, including sulphide veins and stringers containing Cu, Ni, Pt, Pd, and Au, in the brecciated FW rocks beneath the SIC.
- Structurally and/or hydrothermally remobilized sulphide Ni, Cu, Co, Pt, Pd, and Au mineralization.
- Offset dyke deposits, including massive sulphide consisting of Ni, Cu, Co, Pt, Pd, and Au mineralization associated with brecciated and inclusion-bearing phases of the quartz diorite (QD) offset dykes (i.e., inclusion-rich quartz diorite [IQD]).
- Hybrid type deposits representing combinations of the above.

Figure 8-1 shows a cross-section through the SIC contact on the north range, illustrating the host environments for contact and FW mineralization (Lightfoot, 2016).

Deposits of the Crean Hill Project include Contact Type and Footwall Type deposits (see below).

Figure 8-1: Cross-section through the SIC Contact on the North Range (Lightfoot, 2016)



8.1 CONTACT TYPE DEPOSITS

Much of the historic mining activity on the property exploited the first type of deposit mentioned, which are contact type deposits. Mineralization includes blebby to massive accumulations of sulphide, including pyrrhotite > chalcopyrite > pentlandite concentrated within embayment depressions along the base of the SIC, both within the contact sublayer and in the immediately adjacent FW Breccia (though FW Breccia is more prevalent in the North and East Ranges; refer to Figure 8-1).

The massive and semi-massive accumulations of sulphide are strongly conductive and borehole electromagnetics (BHEM) is used routinely on all drillholes of significant depth. Generally, current BHEM technology can detect an off-hole conductor about the same distance as the median dimension of that conductor, with several practical caveats. Maximum effectiveness requires strong coupling between the loop configuration and the conductor.

8.2 FOOTWALL TYPE DEPOSITS

Examples of recent FW deposit discoveries in the region include the Crean Hill 109 FW Zone, parts of the 9400 Zone, McCreedy East FW deposits at Vale's Coleman Mine (the 148, 153, and 170 orebodies), the FW orebodies at Glencore's Nickel Rim South Mine, and the FW deposits at Vale's Victor and Capre development projects.

Mineralization includes networks of one to ten metre sized massive sulphide veins, stockworks of smaller centimetre to metre sized sulphide veinlets, and low sulphide alteration zones with weak sulphide disseminations, including chalcopyrite > pentlandite ± pyrrhotite, millerite, cubanite, bornite, and pyrite. FW deposits are often hosted by Sudbury Breccia structures.

Low sulphide High Grade Precious Metals ("LSHPM") is a relatively new classification of mineralization in Sudbury (Farrow et al., 2005; Péntek et al., 2008; Tuba et al., 2010; Kjarsgaard & Ames, 2010). LSHPM mineralization has been identified as follows in three geological settings:

- Fine-grained specks in FW shears, such as those observed in the 109 FW and 9400 Zones at Crean Hill.
- Fine-grained specks, disseminations, and narrow discontinuous fracture fillings in Sudbury Breccia and adjacent wall rocks in the 109 FW and 9400 Zones at Denison, and at several occurrences in the North Range and East Range of the Sudbury Structure.
- Fine disseminations and specks in QD dykes, lenses, and pods.

The LSHPM mineralization at Crean Hill exhibits a close spatial relationship to the more massive contact-related Ni-Cu sulphide ores at the base of the SIC. This relationship results in the greatest concentration of LSHPM mineralization occurring adjacent to the largest concentration and highest tenor massive sulphide occupying the Crean Hill embayment structure (Lightfoot, 2017).

Sulphide veins within FW deposits are variably conductive and chargeable. Airborne, ground, and Bore Hole Electro Magnetic (BHEM)—as well as ground and borehole DCIP surveys—can be effective in directly detecting the sulphide veins. However, due to the potentially small physical size of individual conductive veins and the low-sulphide nature of some of the PGE-rich FW deposits, the detectable distance of geophysical techniques may be limited. Exploration requires careful geological mapping to understand structural controls, drilling, and extensive sampling, and recognize SIC-related partial melting features and hydrothermal alteration styles associated with FW systems.

8.3 STRUCTURALLY AND/OR HYDROTHERMALLY REMOBILIZED MINERALIZATION

In some deposits, sulphide has been remobilized into shear zones and related structural traps. Important examples of this type of deposit include those at Garson, Falconbridge, Falconbridge East, and Creighton mines. Several mineralized trends at Crane Hill mimic the underlying shear fabric; because of this, these trends may fall under this deposit type. However, it is unclear whether the mineralization has been remobilized or if the shear zones acted as ground preparation providing pathways for the magmatic melts to follow or is a combination of both without knowing the order of mineralization.

8.4 OFFSET DYKE DEPOSITS

Though not identified at Crean Hill, the potential for the property to host offset dyke deposits exists. Examples of recent offset dyke deposit discoveries in the region include the Kelly Lake deposit within the Copper Cliff offset dyke, and the Totten and Victoria deposits within the Worthington offset dyke.

Mineralization includes massive and semi-massive accumulations of sulphide, including pyrrhotite > chalcopyrite > pentlandite. Sulphide accumulations are associated with and are known to concentrate in structural traps such as vertical or horizontal pinches or terminations in the dyke, bends in the dyke, splays/convergences of dyke branches, along the margins or within “pressure shadows” of large blocks caught up in the dyke, and at intersections of the offset dykes with coarse mafic intrusions in the wall rock. Increased PGEs are typically associated with more fractionated chalcopyrite rich zones within offset dyke deposits, which can extend from the dyke outwards into the surrounding country rock, into adjacent zones of Sudbury breccia, meta-breccia or anatexite.

These structural traps are largely controlled by the geology of the wall rock to the offset dykes (geological units, contacts, and structures). Understanding these wall rocks is crucial to developing and prioritizing drill targets below the depth of penetration of surface geophysics.

Geophysically, offset style deposits are similar to contact style deposits discussed in Section 8.1.

9 EXPLORATION

Since becoming operator of the JV in 2014, Loncan had not conducted any significant surface exploration on the property (SRK, 2020) and as of the effective date of this report, Magna has completed a UTEM 5 surface geophysical survey on the Crean Hill property, however the data has not yet been processed and interpreted. Magna commenced exploration diamond drilling in late 2022 and drilling continues in 2023 (see Section 10) on the property.

10 DRILLING

The drilling information in this section for previous owners of the property is sourced from WSP (2020). A total of 4,009 drillholes totalling 515,664 m (1,691,812 ft)—make up the Crean Hill drillhole dataset prior to Magna acquiring the property in late 2022. Table 10-1 summarizes the number of holes drilled and the total footage by year (WSP, 2020).

Since its acquisition of the Crean Hill property in November 2022, Magna has completed some diamond drilling within the Crean Hill Mine. As of the date of this report, drilling has focused on defining continuity and grade of mineralization (within the current mineral resources), and expansion of the known mineralized zones. Magna has completed 68 surface diamond drill holes for a total of 12,180 m. The results of the diamond drilling completed by Magna to date is described below in section 10.2. Results of the Magna drilling results have not yet been considered in the geological interpretation, and assay results have not been incorporated into the mineral resource estimate.

Table 10-1: Summary of Property Diamond Drillholes by Year

Year	# of Holes	Total Metres
1901	154	11,300
1905	10	1,155
1906	9	899
1908	4	593
1909	4	225
1912	7	206
1915	14	278
1917	13	967
1918	20	1,886
1919	1	65
1937	16	4,773
1938	2	590
1945	14	3,343
1950	6	1,975
1951	40	2,800
1952	73	11,835
1953	69	14,457
1954	28	745
1957	78	6,933
1958	45	3,883
1959	118	12,339
1960	198	31,613

Year	# of Holes	Total Metres
1961	49	8,076
1962	7	2,830
1964	10	293
1965	25	2,680
1966	62	5,878
1967	54	8,962
1968	138	15,963
1969	121	12,178
1970	270	20,689
1971	226	15,701
1972	6	975
1975	98	13,068
1976	241	22,823
1977	236	23,766
1978	29	2,459
1983	20	2,864
1984	16	1,421
1986	18	3,005
1987	79	6,181
1988	34	3,130
1989	17	5,050
1990	99	8,234
1991	92	13,205
1992	110	17,031
1993	189	24,418
1994	158	15,581
1995	130	18,993
1996	95	12,116
1997	62	7,065
1998	33	4,801
1999	10	450
2001	2	557
2002	14	3,009
2004	2	20
2005	17	5,507
2006	16	7,052

Year	# of Holes	Total Metres
2007	21	14,157
2008	23	7,879
2009	30	6,819
2010	56	11,271
2011	3	447
2012	19	2,474
2014	40	8,646
2015	33	15,689
2016	58	13,165
2017	18	6,226
Total	**4,009**	**515,664**

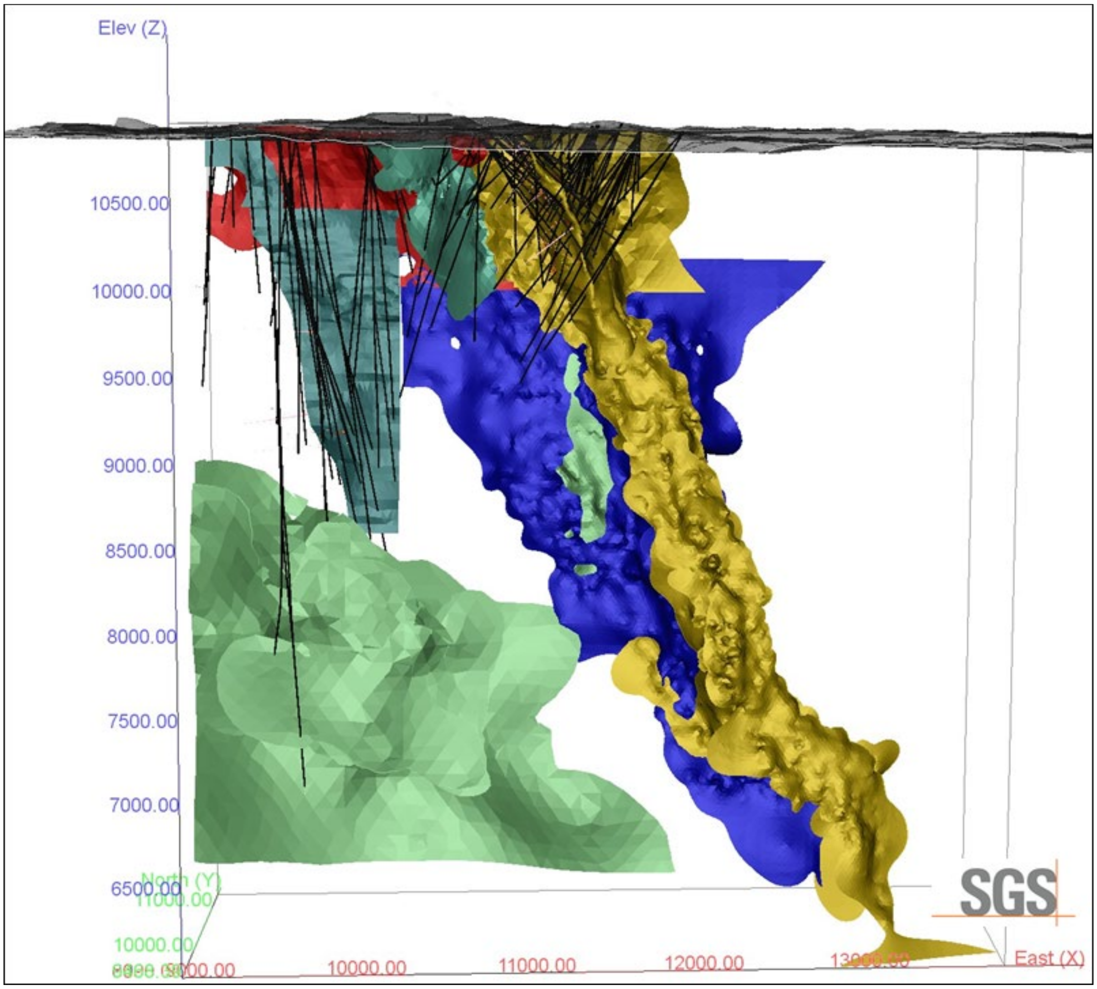
10.1 VALE AND LONCAN DRILLING

10.1.1 Vale Drilling

The diamond drilling by Vale and its predecessor companies dates back to 1906. Drilling equipment has evolved over this period from standard rods to wireline, and core sizes from EX, AQ, BQ, and NQ. The drilling by Vale was focused primarily on supporting the copper-nickel exploration and production at the Crean Hill Mine.

From 2005 until Loncan became the operator of the Project in 2014, the drilling was focused on the LSHPM, as shown in Figure 10-1. During this time, a total of 185 holes totalling 55,605 m (182,430 ft) was completed, of which 176 holes totalling 53,313 m (174,911 ft) targeted the LSHPM.

Figure 10-1: Isometric View Looking North: Distribution of Drill Holes Completed by Vale within the Deposit Area (2005–2014)



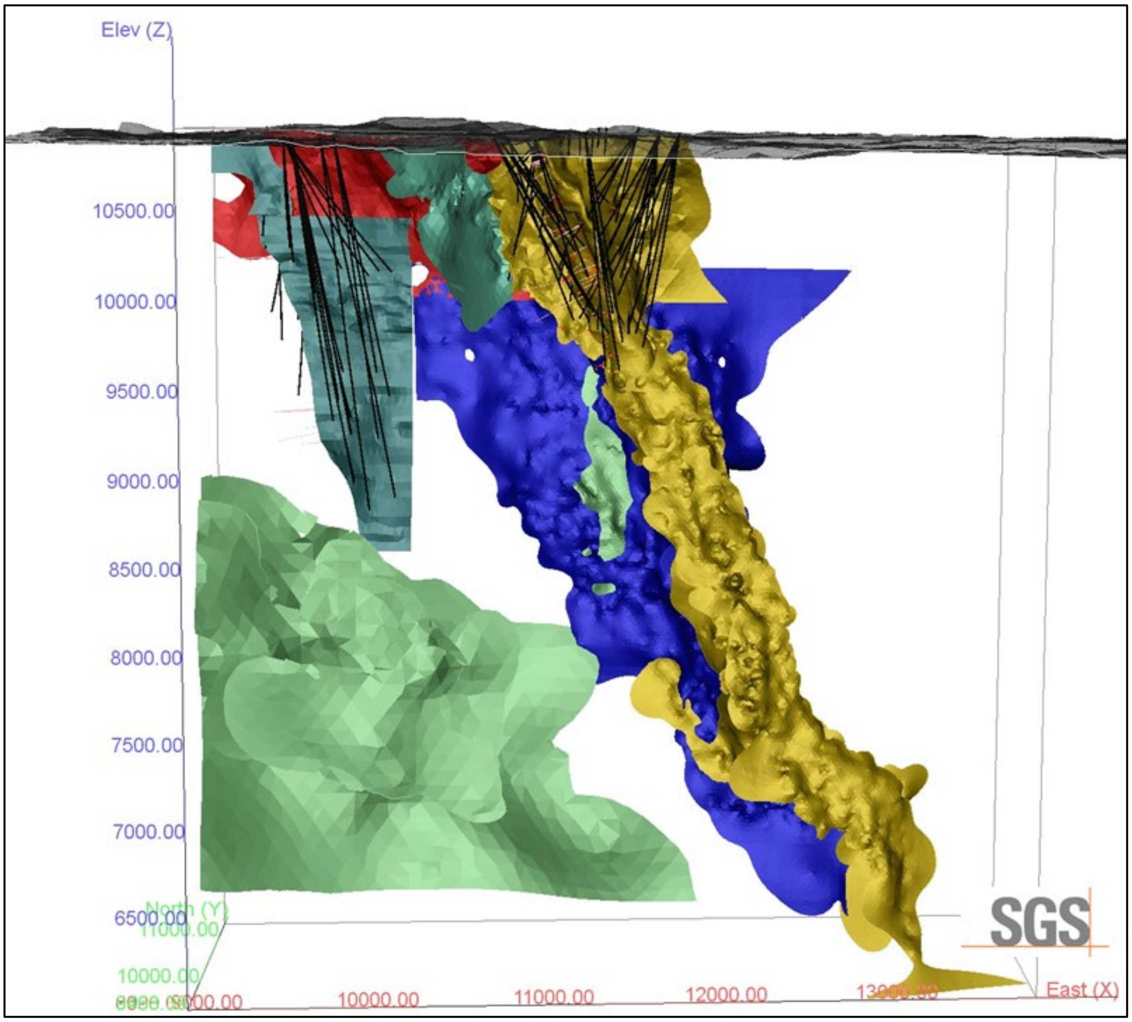
Core recovery tended to be greater than 95%. Recovery losses tended to be near surface in fractured ground or near old underground workings.

The true thickness of core intersections was variable depending on whether the hole was collared on the hanging wall or footwall side of the mineralization and the dip of the hole.

10.1.2 Loncan Drilling

Loncan took over operatorship of the Project in 2014. A total 149 holes totalling 43,726 m (143,458 ft) were completed from 2014 to 2017, as shown in Figure 10-2.

Figure 10-2: Isometric View Looking North: Distribution of Drill Holes Completed by Loncan within the Deposit Area (2014–2017)



The drilling was completed by various drilling contractors using industry standard NQ wireline drill rigs. A small proportion of these holes may be outside the revised property boundary.

All the holes drilled by Loncan targeted the LSHPM material on the Project. Core recovery tended to be greater than 95%. Recovery losses tended to be near surface in fractured ground or in proximity to old underground workings.

The true thickness of core intersections was variable depending on whether the hole was collared on the hanging wall or footwall side of the mineralization and the dip of the hole.

10.1.3 Surveying

Collar

The earliest drilling programs on the Project used a mine grid to spot holes. The Mines Exploration Borehole System (MEBS), Vale's borehole database system, can convert the various Vale grids into a common coordinate system for export.

There is a risk of inaccuracies related to the earlier grid drilling data not matching true survey data; however, this risk was considered low for the purposes of resource estimation.

From 2000 onwards, drill collars were spotted in the field using a real-time differential GPS and surveyed again with the real-time differential GPS upon completion of the hole or program. Survey results from the GPS were completed in the mine grid coordinate or converted from UTM to mine grid.

Survey lines for underground boreholes were marked on the walls by the Vale survey department with front sights and back sights. A borehole layout was provided to the diamond driller with the hole ID, front sight / back sight, and the dip of the hole. Final collar locations were not surveyed after the completion of the boreholes.

Downhole

Various downhole survey methods were used over the Project life, as follows:

- Acid tests (no azimuth, dip only)
- Reflex
- Gyro
- North-seeking gyro

Acid tests were collected approximately every 30.5 m (100 ft) down the hole. Reflex, gyro, and north-seeking gyro surveys were conducted upon completion of the hole, resulting in continuous downhole survey readings approximately every 0.9 to 1.5 m (3 to 5 ft) down the hole.

Acid tests are inherently less accurate that the other methods mentioned here and should be avoided for future programs.

10.1.4 Core Delivery

Surface

Surface core was delivered to the Vale or Loncan core logging facility—depending on the program operator—by the diamond drillers or Vale core technician every weekday.

Underground

Drill core from underground was secured on pallets. Periodically, the pallets were transported by the Vale mine operations group to the shaft and hoisted to surface. The pallets were then delivered to the Vale core logging facility.

10.1.5 Core Logging

During the 2015–2017 Denison drilling program, diamond drill core was transported from the field at the Crean Hill Project to the Loncan core shack—a distance of about 30 km—by either company personnel or by a drill contractor. The core was inspected for continuity and the correct markings of depths, tagged, and then logged, and sample intervals were marked by Loncan geologists (WSP, 2020).

All borehole data from surface and underground drilling are stored in Vale's MEBS database. In addition, all boreholes drilled by Loncan exist as Excel® files exported from MEBS as a back-up record.

MEBS contains data for modern holes recorded directly in the system and holes that existed on the property before the database—which have been transcribed in the database—dating back to 1901. Old boreholes often have extremely short interval descriptions, if any. The borehole database used in this resource estimation was downloaded by Alexander (Sandy) Gibson of Vale on 27 April 2017, in Datamine Studio 3, using Vale Ontario Operation's scripts, which export data directly out of MEBS. The measured density data was manually merged with this dataset using the sample identification number as the key field.

Routine geological logging is conducted by suitably qualified geologists and geological technicians, and it is captured in MEBS. All logging is completed in imperial units, as per MEBS requirements. Rock and minor rock codes, Rock Quality Designation (RQD) angle (foliation or significant contact angle), sample number, ore code, estimated Ni + Cu grade, estimated percent sulphide, and a detailed description are recorded for each interval, where applicable, for each field. Additional data uploaded into MEBS includes core photos, geotechnical logs, and measured density data.

Earlier geological logging that pre-dates MEBS followed procedures which diverge in several respects from the existing current Loncan procedure. These differences are not expected to have a material impact on the integrity of the geological interpretation or understanding.

10.2 MAGNA DRILLING – 2022-2023

To date, Magna has completed 68 surface diamond drill holes between Q4 2022 (November) and Q3 2023 for a total of 12,180 m (Table 10-2). A list of significant drill intercepts to date are presented in Table 10-3. Drilling to date has been designed to enhance Magna's understanding of the near surface Intermediate, 101 FW and 109 FW zones both along strike and down-dip of historical mining areas (Figure 10-3 to Figure 10-6).

Highlights:

101 FW Zone

- **MCR-22-003** intersected 3.8% Nickel, 1.8% Copper, & 0.5 g/t Pt+Pd+Au over 10.8 metres, including 3.2% Ni, 8.3% Cu, & 0.5 g/t Pt+Pd+Au over 1.7 metres, and 6.0% Ni, 0.3% Cu, 0.7 g/t Pt+Pd+Au over 5.8 metres

- **MCR-22-005** intersected 4.0 % Ni, 0.7 % Cu, 0.7 g/t Pt+Pd+Au over 31.1 metres, Including 6.5 % Ni, 1.0 % Cu, 0.5 g/t Pt+Pd+Au over 5.0 metres, and 5.7 % Ni, 0.7 % Cu, 0.8 g/t Pt+Pd+Au over 16.1 metres

- **MCR-22-004** intersected:
  - 2.5% Ni, 1.2 %Cu, 0.7 g/t Pt+Pd+Au over 8.1 metres
  - 1.1% Ni, 5.7% Cu, 2.2 g/t Pt+Pd+Au over 4.4 metres
  - 2.4% Ni, 2.7% Cu, 2.8 g/t Pt+Pd+Au over 4.7 metres

- **MCR-23-011** intersected 2.1% Ni, 3.0 % Cu, 1.1 g/t Pt + Pd + Au over 2.2 metres

- **MCR-23-020** intersected 4.3% Ni, 0.4 % Cu, 1.7 g/t Pt + Pd + Au over 0.4 metres and 11.2% Ni, 0.4 % Cu, 0.1 g/t Pt + Pd + Au over 0.4 metres

- **MCR-23-024** intersected 5.1% Ni, 1.3 % Cu, 0.03 g/t Pt + Pd + Au over 0.8 metres

- **MCR-23-041** intersected 3.0% Ni, 0.7% Cu, 1.2 g/t Pt + Pd + Au over 31.6 metres

- **MCR-23-042** intersected 4.2% Ni, 1.4% Cu, 1.0 g/t Pt + Pd + Au over 27.6 metres

109 FW Zone

- **MCR-22-010** intersected 0.4 % Ni, 0.5 % Cu, 7.2 g/t Pt+Pd+Au over 98.3 metres
  - including higher grade intervals grading 0.8 % Ni, 0.8 % Cu, 12.7 g/t Pt+Pd+Au over 44.0 metres, including 3.7 % Ni, 2.8 % Cu, 20.2 g/t Pt+Pd+Au over 7.1 metres.

- **MCR-23-013** intersected 0.2% Ni, 0.6 % Cu, 11.0 g/t Pt + Pd + Au over 95.4 metres,

  - including 0.76% Ni, 1.54% Cu and 32.83 g/t Pt + Pd + Au over 21.8 metres,
  - including 2.4% Ni, 4.3 % Cu, 37.2 g/t Pt + Pd +Au over 5.1 metres,
  - and including 0.6% Ni, 2.3 % Cu, 122.5 g/t Pt + Pd +Au over 2.8 metres

- **MCB-23-018** intersected 0.2% Ni, 0.1 % Cu, 5.4 g/t Pt + Pd + Au over 27.0 metres,
  - including 0.2 % Ni, 0.1% Cu and 11.2 g/t Pt + Pd + Au over 3.9 metres,
  - and 0.4 % Ni, 0.2% Cu and 11.1 g/t Pt + Pd + Au over 5.7 metres

- **MCR-23-019** intersected 0.2% Ni, 0.3 % Cu, 8.0 g/t Pt + Pd + Au over 28.4 metres

- **MCB-23-021** intersected 0.2% Ni, 0.5 % Cu, 7.3 g/t Pt + Pd + Au over 16.9 metres
  - including 0.4 % Ni, 5.3% Cu and 11.0 g/t Pt + Pd + Au over 0.3 metres,
  - and 1.2 % Ni, 7.7% Cu and 12.3 g/t Pt + Pd + Au over 0.3 metres

Intermediate Zone

- **MCR-22-008** intersected 1.2% Ni, 0.8 %Cu, 1.4 g/t Pt+Pd+Au over 17.3 metres and 3.1% Ni, 0.6% Cu, 0.6 g/t Pt+Pd+Au over 3.7 metres

- **MCR-22-009** intersected 1.1% Ni, 0.7 %Cu, 1.2 g/t Pt+Pd+Au over 27.0 metres

105 FW Zone

- **MCR-23-024** intersected 6.0% Ni, 2.2 % Cu, 5.8 g/t Pt + Pd + Au over 2.2 metres

Table 10-2: List of Drill Holes Completed by Magna in 2022 – 2023

BHID	Easting	Northing	Elevation	Azimuth	Dip	Depth (m)
MCR-22-001	472795	5142006	296	198	50	90
MCR-22-002	472796	5142006	296	198	66	102
MCR-22-003	472897	5142029	304	163	61	237
MCR-22-004	472897	5142029	304	181	52	246
MCR-22-005	472897	5142029	304	161	56	198
MCR-22-006	472898	5142029	304	156	66	249
MCR-22-007	473091	5142012	294	237	49	236
MCR-22-008	473091	5142012	294	244	45	201
MCR-22-009	473092	5142012	294	213	51	251
MCR-22-010	473030	5141815	288	121	46	204
MCB-23-001	473053	5141781	293	116	38	25
MCB-23-002	473053	5141781	294	89	39	50
MCB-23-003	473052	5141779	294	89	77	51
MCB-23-004	473053	5141781	292	160	38	26
MCB-23-005	473051	5141787	292	91	38	44
MCB-23-006	473051	5141787	292	61	38	38
MCB-23-007	473051	5141787	292	61	60	24
MCB-23-008	473053	5141788	292	32	38	20
MCB-23-009	473046	5141792	292	53	38	22
MCB-23-010	473046	5141792	292	7	63	17
MCB-23-011	473046	5141792	288	335	39	17
MCB-23-012	473037	5141805	288	90	38	22
MCB-23-013	473037	5141805	288	46	38	25
MCB-23-014	473028	5141821	288	120	38	50
MCB-23-015	473028	5141821	288	80	38	26
MCB-23-016	473027	5141821	287	10	38	17
MCB-23-017	473079	5141769	297	340	38	35
MCB-23-018	473079	5141769	297	303	39	50
MCB-23-019	473079	5141769	297	270	38	30

BHID	Easting	Northing	Elevation	Azimuth	Dip	Depth (m)
MCB-23-020	473079	5141769	297	270	55	18
MCB-23-021	473080	5141770	297	320	47	51
MCB-23-022	473070	5141754	294	350	42	25
MCB-23-023	473070	5141754	294	310	38	36
MCB-23-024	473061	5141726	293	252	42	80
MCB-23-025	473061	5141726	293	305	38	51
MCB-23-026	473048	5141780	292	265	38	30
MCR-23-011	473030	5141816	288	330	58	413
MCR-23-012	473117	5141716	298	313	59	791
MCR-23-013	473047	5141784	292	102	68	786
MCR-23-014	474203	5141966	300	221	61	150
MCR-23-015	474204	5141965	300	202	46	98
MCR-23-016	474202	5141968	300	241	46	76
MCR-23-017	474207	5141967	300	154	80	76
MCR-23-018	474206	5141966	300	176	55	71
MCR-23-019	473169	5141743	308	300	45	200
MCR-23-020	473031	5141814	288	332	68	380
MCR-23-021	473030	5141814	288	316	55	335
MCR-23-022	472580	5141884	286	357	70	503
MCR-23-023	472585	5141882	286	332	47	440
MCR-23-024	473033	5141811	288	345	72	533
MCR-23-025	472816	5141363	286	284	60	731
MCR-23-026	474330	5142140	296	192	52	350
MCR-23-027	474511	5141877	294	175	44	325
MCR-23-028	474510	5141876	294	100	45	80
MCR-23-029	473130	5141627	296	270	45	350
MCR-23-030	472656	5141369	282	98	50	677
MCR-23-031	472683	5141752	285	91	45	151
MCR-23-032	472985	5141759	281	315	48	299
MCR-23-033	473426	5141934	299	130	59	147
MCR-23-034	472985	5141759	281	323	47	308

BHID	Easting	Northing	Elevation	Azimuth	Dip	Depth (m)
MCR-23-035	473026	5141821	286	333	63	450
MCR-23-036	472984	5141759	281	307	51	299
MCR-23-037	472984	5141760	281	333	53	383
MCR-23-038	473032	5141815	288	325	72	501
MCR-23-039	473099	5141551	301	353	49	575
MCR-23-040	473033	5141813	288	1	72	402
MCR-23-041	472897	5142028	303	158	59	221
MCR-23-042	472897	5142028	303	164	59	200

Table 10-3: Significant Drill Intercepts from the 2022 – 2023 Drilling by Magna

Drillhole	Zone		From (m)	To (m)	Length (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
MCR-22-001	Intermediate		46.94	53.54	6.60	0.51	0.49	0.02	0.14	0.04	0.06
		and	58.00	69.78	11.78	0.53	0.77	0.02	0.33	0.14	0.22
		including	63.63	66.16	2.53	0.81	2.36	0.03	0.63	0.17	0.80
		and	71.40	73.82	2.42	0.23	0.29	0.01	0.40	0.06	0.08
MCR-22-002	Intermediate		51.97	80.29	28.32	0.61	0.46	0.02	0.35	0.09	0.07
		including	61.42	66.82	5.40	1.25	0.38	0.04	0.73	0.16	0.10
MCR-22-003	Intermediate		75.83	105.06	30.23	0.68	0.43	0.02	0.36	0.13	0.11
		including	95.55	105.06	10.51	1.11	0.61	0.03	0.86	0.31	0.20
	101 FW		154.53	157.20	2.67	2.57	0.40	0.07	0.94	0.67	0.17
	101 FW		168.12	178.90	10.78	3.75	1.74	0.09	0.20	0.22	0.04
		including	168.12	169.78	1.66	3.24	8.30	0.08	0.07	0.24	0.21
		and	173.10	178.90	5.80	6.01	0.33	0.14	0.35	0.33	0.02
	101 FW		192.27	195.47	3.20	0.80	3.76	0.06	1.00	2.52	0.67
	101 FW		215.75	222.00	6.25	0.19	0.71	0.01	0.02	0.02	0.08
MCR-22-004	Intermediate		91.55	110.02	18.47	0.70	0.67	0.02	0.28	0.10	0.10
		including	106.77	109.25	2.48	1.33	0.59	0.03	0.96	0.16	0.14
	101 FW		117.61	132.00	14.39	1.65	0.86	0.05	0.32	0.14	0.07
		including	119.06	127.15	8.09	2.46	1.15	0.07	0.48	0.18	0.08
	101 FW		136.61	145.76	9.15	0.81	0.28	0.03	0.09	0.05	0.02
		including	139.65	141.21	1.56	2.65	0.14	0.08	0.13	0.10	0.01
	101 FW		172.83	180.93	8.10	0.21	0.66	0.01	0.11	0.10	0.04
	101 FW		223.58	228.00	4.42	1.11	5.73	0.03	1.17	0.45	0.56
		including	225.55	228.00	2.45	1.17	8.96	0.04	0.99	0.58	1.05
	101 FW		234.29	239.00	4.71	2.43	2.73	0.05	0.24	0.30	2.29
MCR-22-005	Intermediate		75.27	98.09	22.82	0.50	0.48	0.01	0.18	0.06	0.07
		including	88.17	92.23	4.06	0.88	0.97	0.02	0.45	0.16	0.14

Drillhole	Zone		From (m)	To (m)	Length (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
	101 FW		138.39	169.44	31.06	4.04	0.69	0.10	0.36	0.25	0.07
		including	138.39	143.42	5.03	6.50	1.03	0.17	0.28	0.17	0.03
		and	153.08	169.44	16.07	5.68	0.69	0.13	0.43	0.29	0.05
	101 FW		178.70	184.74	6.04	0.19	0.22	0.01	0.38	0.54	0.42
MCR-22-006	Intermediate		90.00	133.76	43.76	0.41	0.43	0.01	0.29	0.09	0.10
		including	125.02	127.51	2.49	1.08	0.95	0.03	1.04	0.17	0.20
	101 FW		137.56	154.82	17.26	0.80	0.77	0.03	0.19	0.12	0.06
		including	142.89	144.83	1.94	2.03	0.56	0.06	0.32	0.26	0.03
	101 FW		160.60	189.00	28.40	0.52	0.67	0.02	0.11	0.09	0.10
		including	166.29	167.74	1.45	2.20	0.22	0.06	0.09	0.07	0.02
	101 FW		208.63	211.64	3.01	0.51	0.45	0.02	0.07	0.08	0.03
MCR-22-007	Intermediate		142.20	200.40	58.20	0.68	0.60	0.02	0.32	0.18	0.11
		including	179.74	184.87	5.13	1.40	1.22	0.05	0.56	0.30	0.14
		and	194.50	198.15	3.65	2.21	0.56	0.07	0.14	0.11	0.08
MCR-22-008	Intermediate		159.00	195.43	36.43	1.04	0.55	0.03	0.49	0.21	0.14
		including	171.88	189.18	17.30	1.24	0.76	0.04	0.83	0.34	0.23
		and	191.69	195.43	3.74	3.07	0.57	0.09	0.33	0.20	0.03
MCR-22-009	Intermediate		175.83	202.78	26.95	1.13	0.69	0.04	0.91	0.20	0.09
MCR-22-010	109 FW		26.55	124.81	98.26	0.39	0.49	0.01	3.36	2.28	1.59
		including	75.85	119.85	44.00	0.77	0.78	0.01	5.79	3.98	2.95
		including	94.31	101.40	7.09	3.70	2.82	0.06	4.53	8.95	6.70
MCR-23-011	Undefined		199.65	200.73	1.08	0.42	0.46	0.01	5.15	5.97	4.01
	101 FW	*and*	279.76	282.00	2.24	2.05	2.98	0.05	0.36	0.71	0.06
	Intermediate	*and*	383.99	386.33	2.34	0.95	1.02	0.04	0.61	0.14	0.14
MCR-23-012	Intermediate		721.66	722.79	1.13	0.96	1.93	0.04	0.49	0.64	0.05
MCR-23-013	109 FW		8.96	104.35	95.39	0.24	0.60	0.01	7.30	2.06	1.66

Drillhole	Zone		From (m)	To (m)	Length (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
		Including	8.96	30.79	21.83	0.76	1.54	0.01	23.55	4.65	4.63
		Including	11.43	16.50	5.07	2.39	4.33	0.04	23.10	8.69	5.36
		and Including	28.00	30.79	2.79	0.57	2.27	0.01	102.92	6.80	12.73
		and	117.00	120.84	3.84	0.03	0.05	0.00	12.23	2.19	0.93
	109 FW Deep		746.58	750.01	3.43	0.10	1.39	0.01	0.00	0.10	0.07
MCR-23-014	G1		13.77	48.28	34.51	0.39	0.39	0.02	0.04	0.04	0.06
		including	13.77	19.00	5.23	1.02	0.76	0.06	0.03	0.07	0.11
		including	45.93	48.28	2.35	0.74	0.33	0.04	0.18	0.15	0.03
MCR-23-015	G1		*No significant values*								
MCR-23-16	G1		0.65	4.42	3.77	0.31	0.18	0.02	0.04	0.06	0.02
MCR-23-17	G1		28.80	31.16	2.36	0.43	0.43	0.02	0.02	0.06	0.05
		and	54.13	66.78	12.65	0.46	0.31	0.02	0.10	0.03	0.03
		Including	62.20	66.78	4.58	0.86	0.27	0.04	0.14	0.05	0.01
MCR-23-18	G1		*No significant values*								
MCR-23-19	109FW		116.59	144.95	28.36	0.15	0.30	0.00	4.53	2.04	1.41
MCR-23-020	109 FW		18.72	19.35	0.63	0.03	0.06	0.00	6.78	0.49	0.30
	105 FW		272.29	272.60	0.31	1.38	0.08	0.07	0.01	1.46	0.13
	101 FW		326.56	327.00	0.44	4.34	0.20	0.23	0.02	0.08	1.59
		and	344.92	354.73	9.81	0.77	0.51	0.01	0.03	0.17	0.30
		Including	347.57	348.00	0.43	11.21	0.68	0.08	0.01	0.06	0.06
		and	362.00	362.60	0.60	0.82	2.05	0.03	0.01	0.01	0.06
MCR-23-021	101 FW		262.74	263.10	0.36	3.17	0.51	0.03	0.12	0.09	1.20
MCR-23-022	9400 FW		338.60	341.02	2.42	0.06	0.07	0.00	3.89	4.70	1.54

Drillhole	Zone		From (m)	To (m)	Length (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
MCR-23-023	9400 FW		*No Significant Values*								
MCR-23-024	105 FW		265.48	270.55	5.07	2.60	0.50	0.05	2.32	1.02	0.04
		Including	265.48	267.63	2.15	6.01	0.97	0.10	5.40	0.32	0.06
	101 FW		333.57	335.23	1.66	2.42	1.04	0.08	0.02	0.05	0.12
		Including	334.00	334.77	0.77	5.06	1.33	0.16	0.01	0.01	0.01
	Intermediate		481.23	483.00	1.77	0.81	0.63	0.03	0.70	0.75	0.27
MCR-23-025	FW Expl		185.90	186.30	0.40	0.01	0.02	0.00	0.01	0.01	1.74
MCR-23-026	G1		*Assays Pending*								
MCR-23-027	G2		*Assays Pending*								
MCR-23-028	G2		*Assays Pending*								
MCR-23-029	FW Expl		*No Significant Values*								
MCR-23-030	FW Expl		*No Significant Values*								
MCR-23-031	FW Expl		*No Significant Values*								
MCR-23-032	101 FW		231.00	234.50	3.50	0.34	0.18	0.02	2.57	1.17	0.06
			237.60	238.30	0.60	2.69	2.68	0.12	0.43	4.05	0.28
MCR-23-033	Intermediate		*Assays Pending*								
MCR-23-034	101 FW		224.30	225.10	0.80	4.58	2.15	0.09	0.41	0.26	0.34
	101 FW		231.40	231.80	0.30	3.25	0.66	0.09	0.11	0.30	0.09
	101 FW		260.20	260.70	0.50	0.47	3.57	0.01	0.46	0.04	0.13
	101 FW		263.40	264.20	0.80	1.92	7.23	0.06	0.01	0.80	0.11
	101 FW		270.20	273.80	3.60	0.21	1.24	0.01	0.01	0.11	0.05
	101 FW		278.60	279.00	0.40	1.26	0.25	0.04	0.01	0.65	0.91
MCR-23-035	101 FW		297.80	300.80	2.90	0.56	2.35	0.02	0.06	0.22	0.39

Drillhole	Zone		From (m)	To (m)	Length (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
	Intermediate		418.20	423.30	4.90	0.84	0.89	0.03	0.88	0.28	0.30
MCR-23-036	101 FW		245.40	246.00	0.50	0.19	1.08	0.00	0.09	0.01	0.10
MCR-23-037	Undefined		92.00	93.40	1.40	2.18	0.88	0.03	0.01	0.01	0.01
	101 FW		309.10	312.80	3.70	0.91	1.87	0.01	0.16	0.05	0.04
MCR-23-038	101 FW		*No Significant Values*								
MCR-23-039			*Assays Pending*								
MCR-23-040			*Assays Pending*								
MCR-23-041	Intermediate		77.40	112.00	34.70	0.60	0.55	0.02	0.33	0.10	0.09
		Including	87.40	95.30	7.90	0.78	0.65	0.02	0.45	0.15	0.10
		and including	102.50	112.00	9.50	0.89	0.85	0.03	0.53	0.15	0.13
	101 FW		134.80	166.50	31.60	2.99	0.65	0.08	0.75	0.37	0.10
MCR-23-042	Intermediate		74.00	80.90	6.90	0.31	0.25	0.01	0.08	0.02	0.04
			90.90	92.60	1.70	0.92	1.33	0.03	0.44	0.30	0.27
	101 FW		149.20	176.90	27.60	4.20	1.37	0.10	0.56	0.36	0.11
MCB-23-001	109 FW		4.00	5.00	1.00	0.06	0.14	0.01	3.92	1.35	0.53
		and	14.00	14.90	0.90	0.08	0.07	0.01	2.98	1.61	0.55
		and	18.18	21.00	2.82	0.05	0.06	0.00	1.28	0.92	0.44
MCB-23-002	109 FW		3.50	7.00	3.50	0.05	0.10	0.01	0.93	0.51	0.33
		and	14.00	14.83	0.83	0.12	0.18	0.01	4.52	3.08	1.44
		and	21.00	22.91	1.91	0.10	0.35	0.01	3.56	2.11	1.16
MCB-23-003	109 FW		8.95	39.95	31.00	0.74	0.58	0.02	3.25	3.27	2.17
		including	25.76	28.94	3.18	5.89	1.99	0.10	2.33	7.19	2.82
MCB-23-004	109 FW		14.32	15.22	0.90	0.10	0.12	0.01	2.66	2.22	1.10

Drillhole	Zone		From (m)	To (m)	Length (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
		and	19.00	20.12	1.12	0.11	0.12	0.01	1.02	0.65	0.41
MCB-23-005	109 FW		14.98	17.00	2.02	0.10	0.16	0.01	1.34	2.08	0.61
MCB-23-006	109 FW		1.85	20.00	18.15	0.12	0.21	0.01	2.42	0.93	0.85
		including	16.14	20.00	3.86	0.18	0.24	0.01	4.53	2.26	1.71
MCB-23-007	109 FW		3.00	24.00	21.00	0.25	0.80	0.01	6.65	1.41	0.89
		including	22.12	24.00	1.88	1.51	6.82	0.03	45.87	1.27	1.92
MCB-23-008	109 FW		13.01	19.43	6.42	0.13	0.24	0.01	3.47	1.07	0.66
MCB-23-009	109 FW		3.00	22.00	19.00	0.23	0.23	0.01	5.52	4.30	2.56
MCB-23-010	109 FW		9.03	12.00	2.97	0.12	0.07	0.01	2.49	1.79	0.92
		and	14.95	16.99	2.04	0.08	0.17	0.01	3.15	0.85	0.58
MCB-23-011	109 FW		8.95	17.00	8.05	0.13	0.31	0.00	2.87	1.57	0.96
MCB-23-012	109 FW		8.04	21.00	12.96	0.36	0.56	0.04	2.12	15.17	1.58
		Including	15.00	19.94	4.94	0.80	1.07	0.10	3.00	38.50	3.41
MCB-23-013	109 FW		11.72	12.40	0.68	0.05	0.47	0.01	0.06	0.17	0.10
MCB-23-014	109 FW		44.37	50.00	5.63	0.13	0.38	0.01	3.30	1.84	1.56
MCB-23-015	109 FW		15.00	26.00	11.00	0.07	0.46	0.01	0.60	0.44	0.38
		including	18.17	18.49	0.32	0.48	8.04	0.04	2.73	5.60	3.95
MCB-23-016	109 FW		*No Significant Values*								
MCB-23-017	109 FW		10.50	11.00	0.50	0.06	0.03	0.01	3.34	1.22	0.71
MCB-23-018	109 FW		20.98	48.00	27.02	0.15	0.12	0.01	2.67	1.73	0.99
		including	20.98	24.87	3.89	0.18	0.11	0.01	5.11	3.98	2.09
		and	32.00	37.65	5.65	0.36	0.17	0.01	5.48	3.60	2.06

Drillhole	Zone		From (m)	To (m)	Length (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
MCB-23-019	109 FW		14.00	17.98	3.98	0.09	0.07	0.01	3.46	2.59	1.25
		and	28.00	29.00	1.00	0.09	0.08	0.01	11.50	2.33	1.68
MCB-23-020	109 FW		*No Significant Values*								
MCB-23-021	109 FW		32.06	49.00	16.94	0.15	0.45	0.01	3.32	2.55	1.41
		including	37.82	38.12	0.30	0.41	5.34	0.01	3.74	4.07	3.17
		and	42.88	43.19	0.31	1.23	7.65	0.05	2.21	7.30	2.81
MCB-23-022	109 FW		*Assays Pending*								
MCB-23-023	109 FW		*Assays Pending*								
MCB-23-024	109 FW		*Assays Pending*								
MCB-23-025	109 FW		*Assays Pending*								
MCB-23-026	109 FW		*Assays Pending*								

Figure 10-3: Plan View of the Crean Hill Property Showing the Location of the 101 FW and 109 FW Zones (Source – Magna Mining)

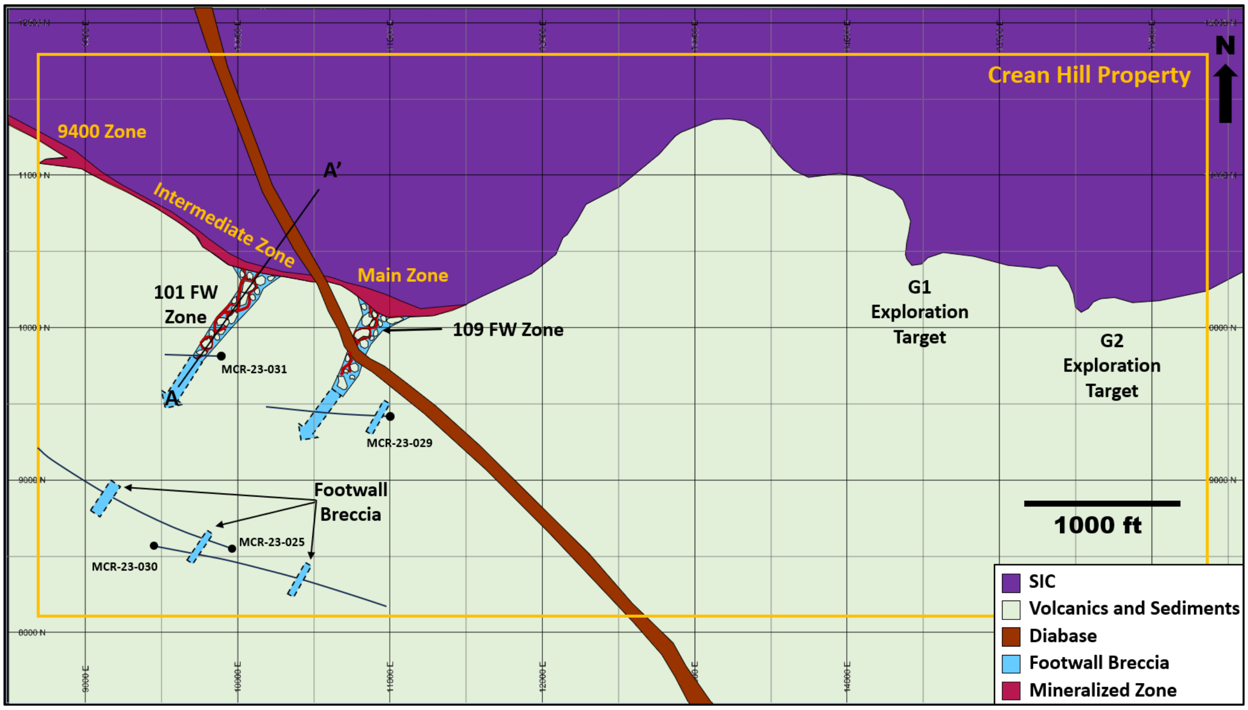


Figure 10-4: Plan View of the 109 FW Zone, Showing the Location of Near Surface Drillholes (Source – Magna Mining)

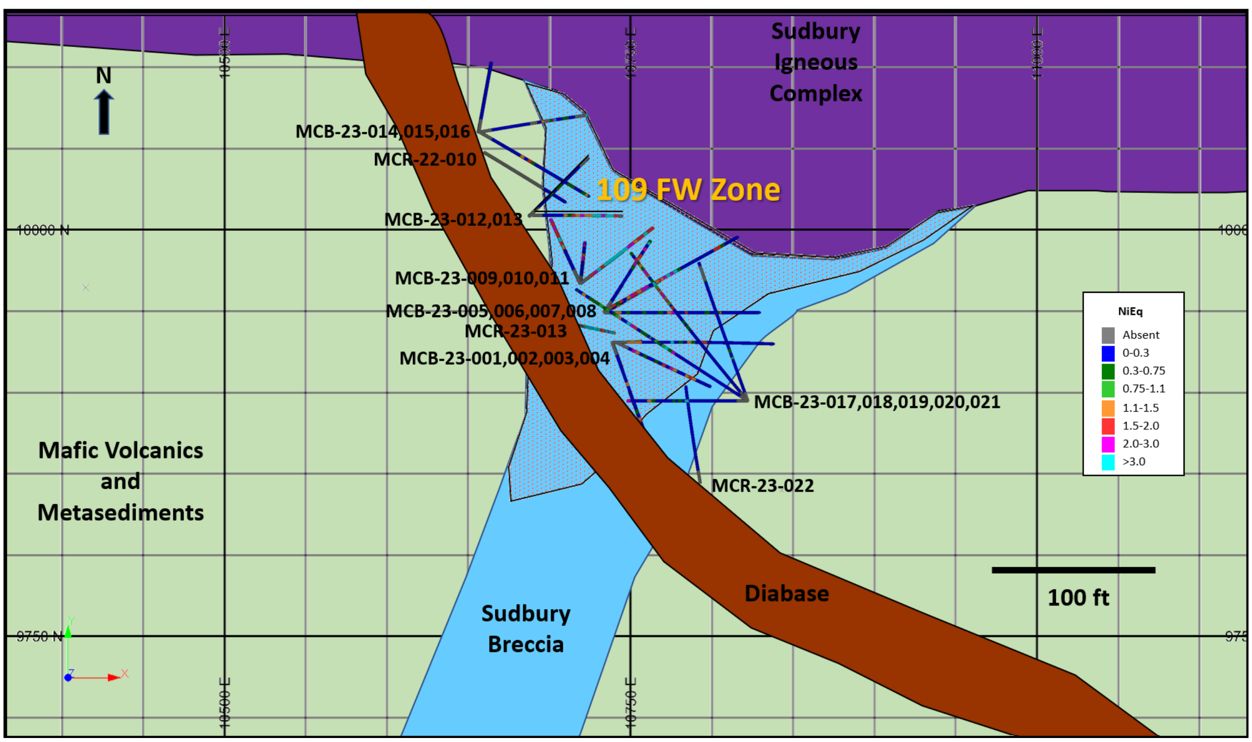


Figure 10-5: Longitudinal Section of the Crean Hill 101 Footwall Zone. Magna Diamond Drillholes and the Location of Section B (Figure 2) are Illustrated. Refer to Figure 10-3 for the Location of Longitudinal Section A-A' (Source – Magna Mining)

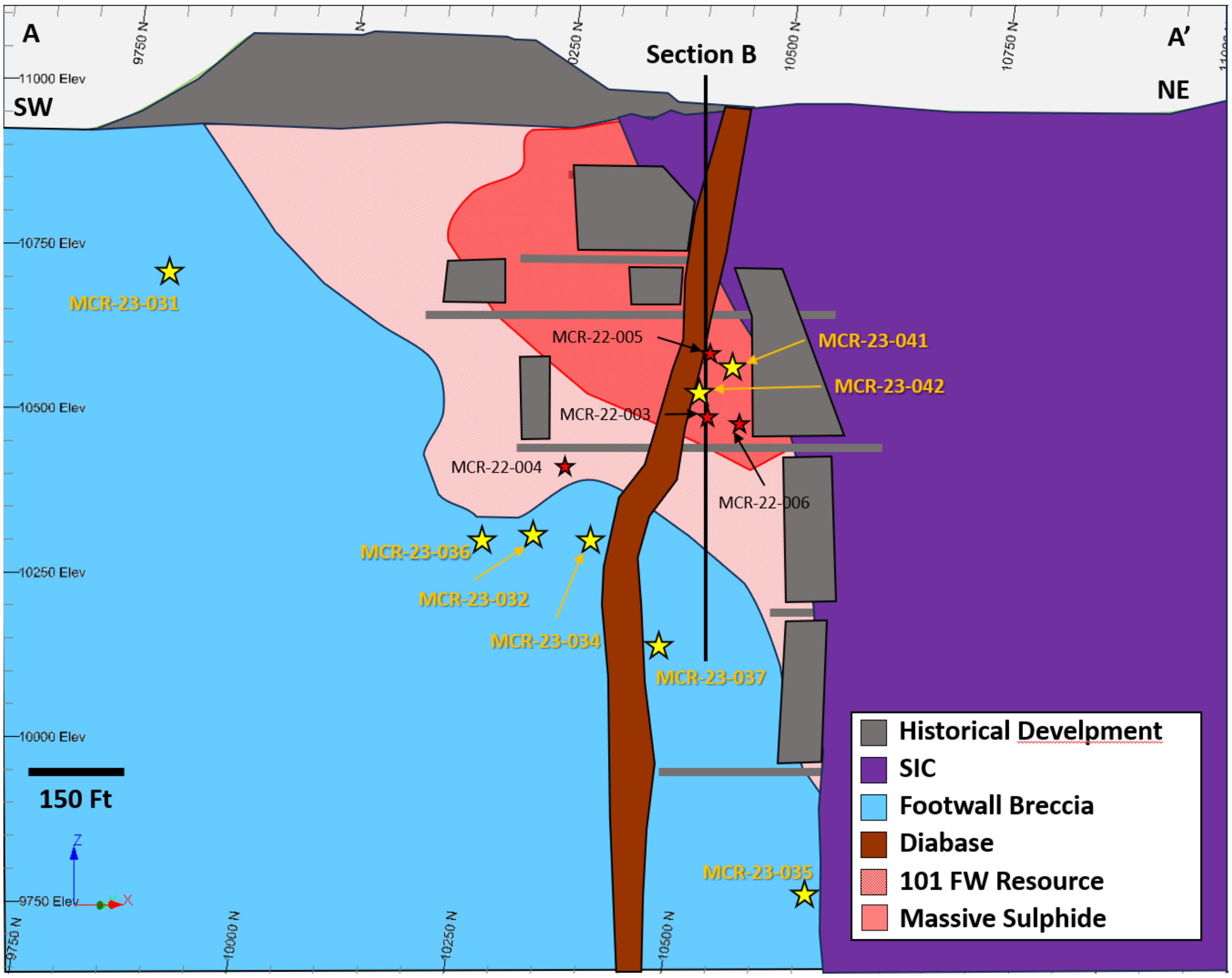
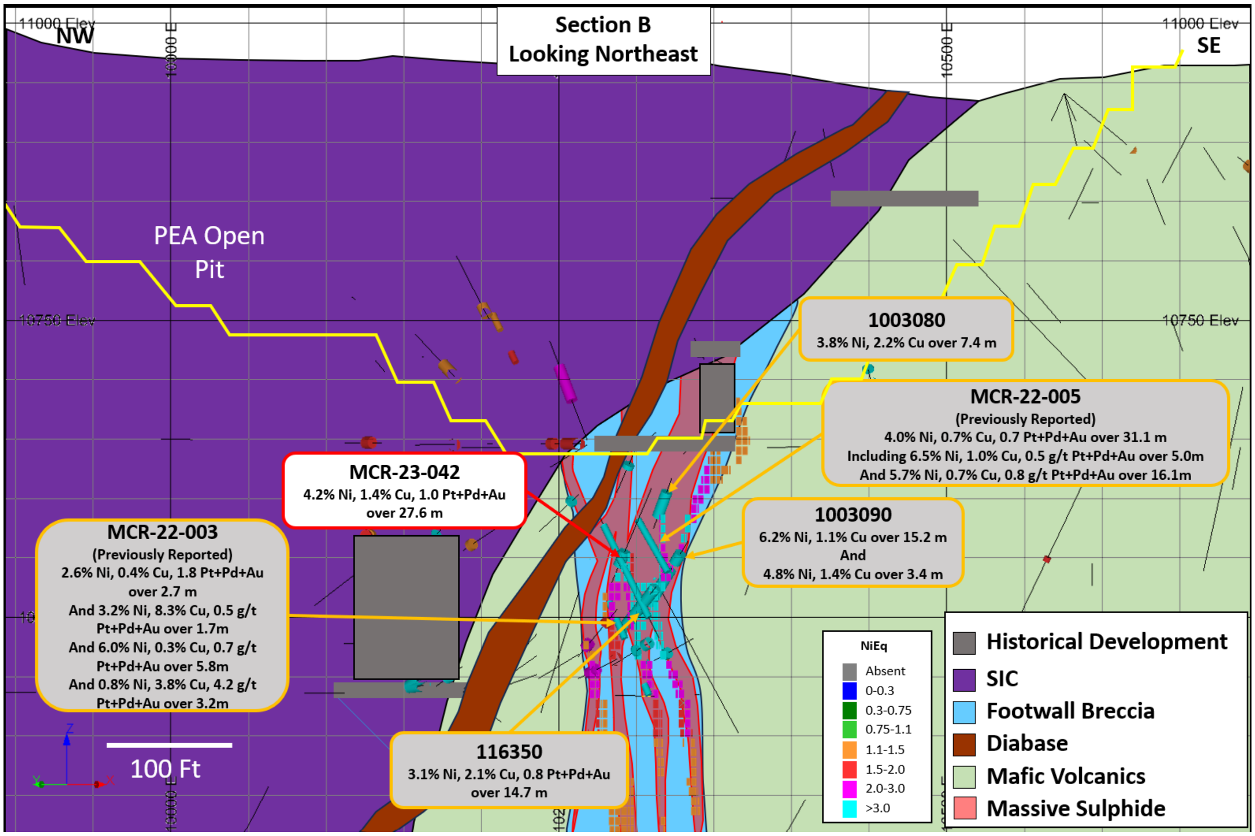


Figure 10-6: Vertical Section B, through the 101 FW Zone, Showing General Geology, Recent and Historical Drilling Results, and the Current Resource Model. Refer to Figure 10-5 for location of Section B (Source – Magna Mining)



11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Magna mining commenced diamond drilling on the Project in Q4 2022, and drilling continues as of the effective date of this report. Results of the Magna drilling have not yet been considered in the geological interpretation, and assay results have not been incorporated into the mineral resource estimate. The 2022-2023 data has not been reviewed by the Author. The effective date of the mineral resource remains 19 August 2022.

The information in this section regarding sample preparation, analyses, and security tasks completed by previous property operators is sourced from WSP (2020), and Baker and Hoffman (2015; 2017).

11.1 CORE SAMPLING

The sampling of cores varied with the age and focus of the drilling program. Older Vale programs, particularly the underground programs focused on definition drilling, used whole core sampling. Surface exploration programs used a rock saw to cut the core in half, with one half of the core placed in sample bags with the appropriate sample tag, and the other half of the core returned to the core box. The core logging and assay methods used by Vale in earlier drilling campaigns are described in Appendix A.

Drill core sampling is guided by lithology, alteration, or visible mineralization. However, due to the nature of the low sulphide mineralization at Denison, sampling is routinely extended over the entire expected mineralized interval and extended to a wider buffer zone on either side. The sampling interval is continuous, with no gaps left where mineralization is perceived to be absent. The maximum sample length is 1.5 m (5 ft) and the minimum, 30 cm (6”). To facilitate compatibility of the data with Vale’s MEBS program, marking, logging and sampling are done in imperial rather than metric units. Every effort is made so that that the sample sent to the laboratory is representative of the entire section of core; however, due to nugget effects and the heterogeneity that is common with this type of PGE mineralization, it is not guaranteed that an assay could be repeated. The half-core samples selected for assay are un-orientated. All samples are sealed (i.e., stapled) in individual, labelled plastic bags with a unique sample tag.

Cores are halved using a water-cooled diamond saw that is cleaned regularly to avoid sample-to-sample contamination. One half of each core is submitted to the lead laboratory—ALS Minerals in Sudbury—by Loncan staff for analysis; the other half is retained on outdoor, roofed core racks at the Loncan office at 129 Fielding Road in Lively, Ontario, as a representative sample or for possible re-sampling. Prior to dispatch to the sample analysis laboratory, each individual sample is weighed.

11.2 SAMPLE PREPARATION

Samples received by ALS laboratory are processed using the sample preparation package PREP-31, as follows:

- Sample is logged into tracking system and a bar code label is attached.
- Dry, crush (<5 kg) 70% passing -8 mesh (2 mm).
- Rotary split (250 g) using a Boyd rotary splitter.

- Pulverize (to 85% passing -75 μm).

At no time has an employee of Loncan been involved in the preparation of the samples.

After the samples are processed, the 250-g splits are then transported by ALS Minerals to their analytical facilities in Vancouver, British Columbia, via courier. Coarse and pulp rejects are retained at the Sudbury facility for a minimum period of six months; however, these are routinely collected by Loncan personnel for storage at the Loncan office facility.

11.3 ANALYTICAL METHOD

Before 2013, all data was acquired by Vale. From 2013 onwards, the data was collected by both Vale and Loncan. The most recent data, from 1999 to 2017, was produced by the ALS Laboratories in Sudbury. ALS is an internationally recognized laboratory accredited by the Standards Council of Canada (SCC) for specific tests listed in ALS's Scopes of Accreditation, which conforms with CAN-P-1579: Requirements for the Accreditation of Mineral Analysis Testing Laboratories; and CAN-P-4E ISO/IEC 17025: General Requirements for the Competence of Testing and Calibration Laboratories. Armitage is independent of ALS.

The analytical assay methodology varied over time, as summarized in Table 11-1.

Table 11-1: Analytical Summary (WSP, 2020)

Period	Dates	Comments
1	Pre–1968	• Values of S (and therefore specific gravity [SG]) based on composite samples. During 2007–2008 all SG were re-calculated using Cu, Ni (and available S) for consistency. • Values of PGE + Au taken infrequently and based on composite samples. • Values of Co based on assay of combined Ni + Co and regression from Ni.
2	1968–1972	• All samples assayed for Cu, Ni, and Co. • PGE + Au and S (SG) as in Period 1 (Pre-1968).
3	1972–1974	• All samples assayed for Cu, Ni, Co, S, and Fe. • PGE + Au as in Period 1 (Pre-1968).
4	1974–1984	• All samples assayed for Cu, Ni, Co, S, and Fe. • Values of PGE + Au taken infrequently, but from individual samples. Values determined using arc-spark emission spectrography.
5	1984–1999	• All samples assayed for Cu, Ni, Co, S, and Fe. • Values of PGE + Au taken infrequently, but from individual samples. Values determined using DCP.
6	1999–Present	• All samples assayed for Cu, Ni, Co, S, Fe, PGE, and Au.

For analysis at ALS, samples underwent the proprietary PGM-ICP23 process, which involves fire assay with standard lead collection of a 30 g aliquot for Pt, Pd, and Au. This is followed by a combination of inductively coupled plasma mass spectrometry (ICP-MS) and inductively coupled atomic emission spectrometry (ICP-AES) to finish the samples.

Samples that exceed 10 g/t on any individual PGE are also run through the PGM-ICP27 process, which recalibrates the ICP-AES finish to accurately report values up to 100 g/t for the PGE. Samples are also analyzed for 33 trace elements and base metals (including Ni, Cu, Co, and Cr) using a four-acid (HNO3-HClO4-HF and HCl) near total digestion, and a combination of ICP-MS and ICP-AES (ME-ICP61 process). ICP-MS over-limits for the ME-ICP61 process are reanalyzed using HF-HNO3-HClO4 acid digestion, HCl leach, and ICP-AES (ME-OG62 process).

In the event of visibly higher-grade mineralization, the preference is to analyze sample groups by submitting the samples directly for the analytical methods described for over-limits, with a specific sample tag prefix. These analytical methods, also referred to as High Grade/Ores Methods, are comprised of HF-HNO3-HClO4 acid digestion, HCl leach, and ICP-AES.

In addition to High Grade/Ores Methods, sulphur is analyzed using Total Sulphur by LECO to accommodate the anticipated higher sulphur levels. ALS Minerals provides assay results to Loncan's Senior Manager, Exploration and Project Geologist via e-mail in MS Excel® format.

11.4 QUALITY ASSURANCE AND QUALITY CONTROL PROGRAMS

No additional QA/QC has been performed by Loncan on the data acquired by Vale because it was accepted that the Vale QA/QC protocol and system is MEBS was adequate. This included all data collected prior to Loncan becoming the property operator in 2014.

After borehole data has been finalized, only the MEBS administrator can make changes or re-classify them as available for changes to be made by another user. All the boreholes used in this estimate have been finalized in this manner. It has been accepted that all the Vale data has been assayed by an accredited laboratory that uses standard reference materials and strict internal QA/QC procedures, and that the data has been adequately reviewed by qualified individuals.

The Certified Reference Materials (CRMs) used on the Denison program from 2014 are PGM standards PGMS-24 and PGMS-25. Subsequently, in 2016, two additional standards—PGMS-27 and PGMS-28—were introduced to the QA/QC process due to the exhaustion of stocks of PGMS-24.

All CRMs were obtained from CDN Laboratories in Vancouver—the first two are sourced from Stillwater (specifically the J-M Reef), both are low Ni-Cu, one is low in 3E (PGMS-25; 2.7 g/t), and one is moderate in 3E (PGMS-24; 6.7 g/t). The second two were made available in 2016 and have a different provenance; PGMS-27 is sourced from the skarn-related Serra Pelada Au-PGE deposit in Brazil, with a moderate 3E grade of 8.09 g/t 3E; and PGMS-28 comes from the low-grade (3.45 g/t 3E) Platreef from the central section of the Bushveld Complex's Northern Limb. In PGMS-27, Au forms 65% of the precious metal assemblage, while it forms only 5% in PGMS-28.

Standards are supplied in batches of one hundred 50-g envelopes via courier from Vancouver. No separate Ni-Cu standards are used. Standards are inserted randomly in the sample order. Blank samples of quartz sand are also inserted in the sample order immediately after an expected high-grade PGE/Ni-Cu sample.

Upon receipt of the assay data from ALS, the samples representing CRMs, blanks, and sample duplicates are highlighted and compiled manually for inspection. Assay values are denominated in Avoirdupois ounces per short ton, and these are converted directly into metric grams per tonne using a conversion factor of 34.28657. The values are then imported into a separate MS Excel® monitoring sheet and plotted graphically.

The QA/QC tolerances and hurdles for the Project are based on the means and standard deviations of the round robin laboratory data for the individual PGMS standards. A batch failure is considered to be when any of the individual 3E assays for a standard sample in the batch exceeds the mean ±3SD threshold, or when more than one sample in a batch exceeds the mean ±2SD threshold on any of the 3E.

The MS Excel® monitoring sheet is conditionally formatted on a True or False basis for each individual precious metal, such that a failure to meet the 3SD criterion is flagged as a False entry and highlighted. The assay values are plotted together on a time-ordered scatter plot graph for each individual envelope with the round robin laboratory data—on which the accepted mean and variance values for the standard are based.

11.5 2014–2015 DRILL PROGRAMS

11.5.1 Borehole Core Sampling and Assay

In the 2014–2015 drilling campaign, core sampled outside the mineral envelope was sampled at 1.5-m intervals; core sampled within the FW and HW mineralisation envelopes, and an approximate 10-m enclosing margin around the envelopes, was sampled at 1 m, except where a geological unit was narrower or to make up the length between a sample above and a geological contact. The minimum sample length was 30 cm (Baker and Hoffman, 2015).

Core was cut in half with a water-cooled saw. One half was placed into a sample bag with a bar-coded sample tag, and the other half was stored at the Lonmin office at 129 Fielding Road, Lively, Ontario, as a representative sample or for possible re-sampling.

Samples were weighed, placed in rice bags for transport, and delivered to ALS Chemex by Lonmin personnel.

11.5.2 Density Data

The majority of the dataset has no measured density values. In previous resource estimations, Vale applied the Alcock formula, as follows, to calculate SG based on assay results:

$$SG = 100 / (100 / 2.88 + 0.0166 * Cu - 0.1077 * Ni - 0.328 * S)$$

This formula was developed for semi-massive- to massive-contact Ni-Cu sulphide deposits. The minimum value possible is 2.88 g/cc, which is known to be too high for some of the host rocks in the 109 FW Domain. A comparison of measured to calculated values revealed the calculated value for felsic rock types (i.e., rhyolite and siliceous metasediment) was approximately 5% too high; for mafic rock types (i.e., metabasalt) the calculated values were approximately 3% too low. There are 4,545 measured values that were logged in the borehole database prior to data capture, a subset of which fell within the mineral envelope.

SG data was collected during the 2014–2015 drilling campaign for most samples in a buffer adjacent to the mineral envelope by measuring the dry and submerged weight. The first several boreholes of this campaign were processed prior to acquiring the SG equipment. Each sample was allowed to fully dry after being cut, weighed on top of the balance, placed in a mesh basket suspended from a free-hanging hook below the balance, and weighed submerged in water. The water was kept at approximately 20°C using a heater/agitator. SG was calculated using the following formula, where 0.998 is a factor to account for the density of water at 20°C:

SG = Dry weight / (Dry weight – Wet weight) * 0.998

Where 0.998 is a factor to account for the density of water at 20°C.

The balance was checked with reference weights, and a reference rock sample had its SG determined prior to each day of SG determinations to produce high quality data. Data was checked for values below 2.5, with one value removed from the dataset.

Vale also collected SG data using the same method. No information was provided on any QA/QC performed for the Vale dataset, but it assumed that the data collected is adequate for use in the resource estimation. Two values below 2.5 were removed from the dataset.

11.5.3 Data Management

All borehole data from surface and underground drilling are stored in Vale's MEBS database. In addition, all boreholes drilled in the 2014–2015 campaign exist as MS Excel® files exported from MEBS to serve as back-up records.

Data—including core photos, geotechnical logs, and SG data—have been uploaded into MEBS, and copies from the 2014–2015 campaign are retained. Copies from all previous drilling campaigns since the inception of the joint venture have been obtained from Vale.

MEBS contains data from holes logged directly in the system and holes that existed previous to the database, dating back to 1901 on the Denison Property. The lithology descriptions for older holes often do not correspond with modern nomenclature.

11.5.4 2014–2015 Borehole Assay QA/QC

All samples were weighed before submitting to ALS Chemex and were re-weighed by ALS upon receipt. All samples were reviewed for consistent weights to identify sample switches. No sample swaps have been identified to date.

All assay data has been reviewed in MEBS's internal QA/QC system to identify samples with weights different from those expected (based on sample length and density calculated using the Alcock formula, as outlined in Section 11.5.2), and to identify sulphide and grade estimates inconsistent with assay results. This routine allows for identification of any blanks or standards that have had sample numbers erroneously entered in place of a core sample, and sample swaps.

A change request routine allows any required changes to be made by the MEBS administrator and records that the change has been made. All corrections requested were completed prior to finalizing holes in the 2014–2015 drilling campaign. No sample swaps were identified with this routine in the 2014–2015 drilling campaign.

Blind CRMs—referred to as standards—and field silica blanks were included in sample runs, and submitted on the basis that the minimum oven batch size at the lab is in the range of 20 to 24 samples. Standard procedure is to include in each batch one CRM, one blank sample, and one sample duplicate, which is two quarter cores from the same half core sample. The CRM was inserted randomly into the batch, while the blank sample was placed in sequence immediately after where the highest PGM grade was expected. The position of the sample duplicate was random. The sample book used to track the samples was in the standard Vale format, using their numbering system to allow for easy integration of assay results into their borehole database.

The ALS Canadian analytical laboratories are accredited by the SCC for specific tests listed in ALS's Scopes of Accreditation, which conforms with CAN-P-1579: Requirements for the Accreditation of Mineral Analysis Testing Laboratories; and CAN-P-4E ISO/IEC 17025: General Requirements for the Competence of Testing and Calibration Laboratories.

Becoming ISO-accredited involves detailed, on-site audits to evaluate ALS's quality management system and verify the technical competence of methods and personnel. This technical verification includes the requirement for successful participation in inter-laboratory proficiency testing programs and full method validation.

At ALS Minerals, received samples are checked against requisition documents prior to being dried, weighed, and then crushed to 70% passing -2mm. They are then Boyd rotary split to 250 g, and this is pulverized and split to better than 85% passing 75 microns. The 250-g splits are then transported by ALS Minerals to their analytical facilities in Vancouver, British Columbia via courier. Coarse and pulp rejects are retained at the Sudbury facility for a minimum period of six months; however, these are routinely collected by Lonmin personnel for storage at the Lonmin office facility.

For routine analysis at ALS, samples undergo the proprietary PGM-ICP23 process, which involves fire assay with standard lead collection of a 30-g aliquot for Pt, Pd, and Au. This is followed by a combination of ICP-MS and ICP-AES to finish the samples.

Samples that exceed 10 g/t on any individual PGE are also run through the PGM-ICP27 process, which recalibrates the ICP-AES finish to accurately report values up to 100 g/t for the PGE. Samples are also analyzed for 33 trace elements and base metals (including Ni, Cu, Co, and Cr) using a four-acid (HNO3-HClO4-HF and HCl) near total digestion, and a combination of ICP-MS and ICP-AES (ME-ICP61 process). ICP-MS over-limits for the ME-ICP61 process are reanalyzed using HF-HNO3-HClO4 acid digestion, HCl leach, and ICP-AES (ME-OG62 process).

In the event of visibly higher-grade mineralization, the preference is to analyze sample groups by submitting the samples directly for the analytical methods described for over-limits, with a specific sample tag prefix. These analytical methods, also referred to as High Grade/Ores Methods, are comprised of HF-HNO3-HClO4 acid digestion, HCl leach, and ICP-AES.

In addition to High Grade/Ores Methods, sulphur is analyzed using Total Sulphur by LECO to accommodate the anticipated higher sulphur levels. ALS Minerals provides assay results to Loncan's Senior Manager, Exploration and Project Geologist via e-mail in MS Excel® format.

Assay results are also provided to Vale's MEBS administrator through their online Webtrieve service, which allows direct importing into their MEBS database.

The Certified Reference Materials (CRMs) used on the Denison program since 2014 are PGM standards PGMS-24 and PGMS-25 obtained from CDN Laboratories in Vancouver—these are sourced from Stillwater (specifically the J-M Reef), both are low Ni-Cu, one is low in 3E (PGMS-25; 2.7 g/t), and one is moderate in 3E (PGMS-24; 6.7 g/t).

Standards are supplied in batches of one hundred 50-g envelopes via courier from Vancouver. No separate Ni-Cu standards are used. Standards are inserted randomly in the sample order. Blank samples of quartz sand are also inserted in the sample order immediately after an expected high-grade PGE/Ni-Cu sample.

Upon receipt of the assay data from ALS, the samples representing CRMs, blanks, and sample duplicates are highlighted and compiled manually for inspection. Assay values are denominated in Avoirdupois ounces per short ton, and these are converted directly into metric grams per tonne using a conversion factor of 34.28657. The values are then imported into a separate MS Excel® monitoring sheet and plotted graphically.

The QA/QC tolerances and hurdles for the Project are based on the means and standard deviations of the round robin laboratory data for the individual PGMS standards. A batch failure is considered to be when any of the individual 3E assays for a standard sample in the batch exceeds the mean ±3SD threshold, or when more than one sample in a batch exceeds the mean ±2SD threshold on any of the 3E.

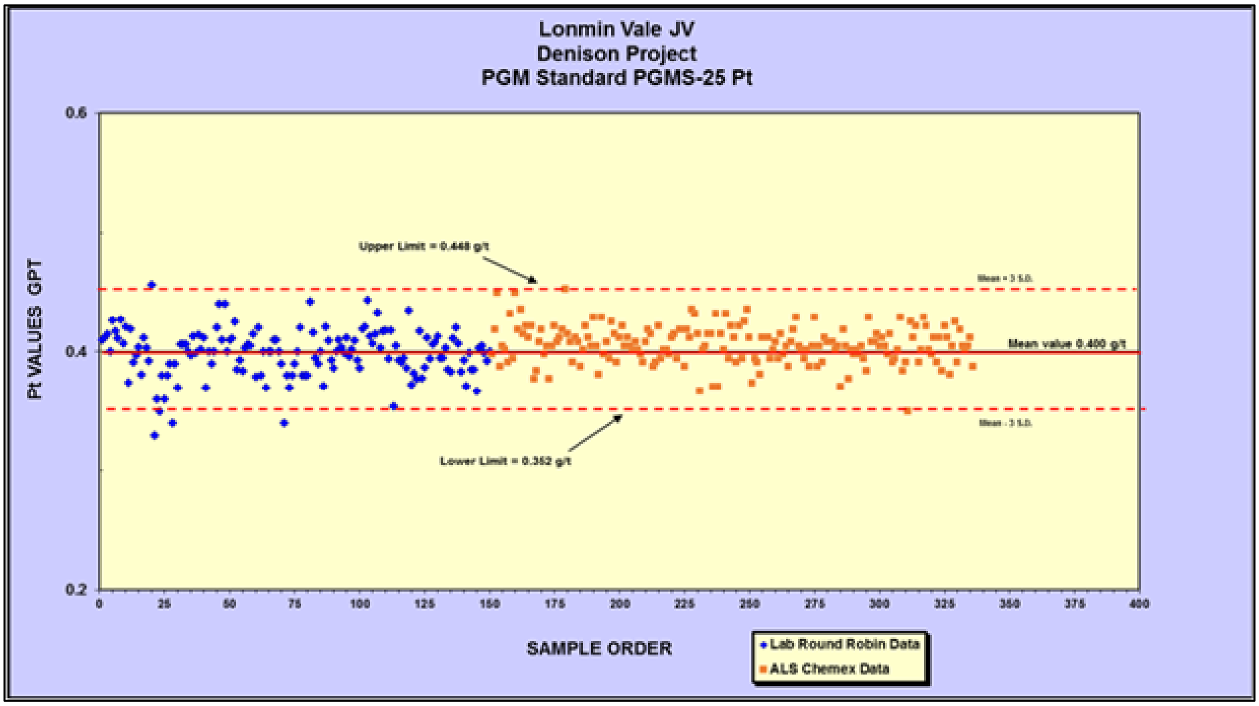
The MS Excel® monitoring sheet is conditionally formatted on a True or False basis for each individual precious metal, such that a failure to meet the 3SD criterion is flagged as a False entry and highlighted; Figure 11-1 represents an example.

Figure 11-1: Example of Tracking Spreadsheet for Certified Reference Material Samples (Baker and Hoffman, 2015)

83	LAB DATA							
84			LAB	LAB	LAB	CHECK	CHECK	CHECK
85	Sample	Lab	ALS Chemex	ALS Chemex	ALS Chemex			
86	#	Work #	g/t Au	g/t Pt	g/t Pd	Au	Pt	Pd
87	R318096	SD14126271	0.463	0.398	1.841	TRUE	TRUE	TRUE
88	R318122	SD14126271	0.483	0.418	1.906	TRUE	TRUE	TRUE
89	R318185	SD14126271	0.483	0.449	1.875	TRUE	FALSE	TRUE
90	R318207	SD14128040	0.535	0.387	1.862	TRUE	TRUE	TRUE
91	R318266	SD14128040	0.463	0.405	1.869	TRUE	TRUE	TRUE
92	R318333	SD14129804	0.477	0.401	1.886	TRUE	TRUE	TRUE
93	R318359	SD14129804	0.470	0.391	1.858	TRUE	TRUE	TRUE
94	R318379	SD14131095	0.482	0.432	1.817	TRUE	TRUE	TRUE
95	R318453	SD14131095	0.459	0.394	1.851	TRUE	TRUE	TRUE
96	MG219087	SD14130901	0.504	0.449	1.917	TRUE	FALSE	TRUE
97	MG219105	SD14130901	0.518	0.418	1.869	TRUE	TRUE	TRUE
98	MG219127	SD14130901	0.501	0.435	1.917	TRUE	TRUE	TRUE
99	MG219193	SD14135517	0.513	0.415	1.910	TRUE	TRUE	TRUE
200	MG219231	SD14135517	0.495	0.422	1.917	TRUE	TRUE	TRUE
201	MG219277	SD14135517	0.499	0.411	1.882	TRUE	TRUE	TRUE
202	MG219297	SD14135517	0.490	0.422	1.810	TRUE	TRUE	TRUE
203	MG219312	SD14138433	0.492	0.377	1.814	TRUE	TRUE	TRUE

The assay values are plotted together on a time-ordered scatter plot graph for each individual envelope with the round robin laboratory data—on which the accepted mean and variance values for the standard are based. The graph (as shown in Figure 11-2) displays an envelope bounded by the *mean + 3SD* and *mean – 3SD* thresholds, and failures lie outside of that envelope.

Figure 11-2: Example of Tracking Graph for Certified Reference Material Samples Showing Mean and Threshold Envelope (Baker and Hoffman, 2015)



The graphs serve to show any overall and between-batch bias for each of the elements in each of the standards, as well as the overall precision. As of the date of this report, there is a small positive variance for each of the individual 3E relative to the round-robin means, but no discernable trend with time. The precision in the ALS data is generally better than that of many of the round-robin labs used to compile the standards.

A batch failure would necessitate repeat assay of the entire batch, from coarse reject stage, with new control samples. For blank sample values, batch failure is more subjective and a failure ceiling value has not been set—very occasionally there have been instances of blank values returning up to 0.24 g/t 3E, but it is likely that values had been carry over from the previous high-grade sample in the prep stage. In these instances, the lab was notified of the issue. Both blank and standard sample insertion also serve to highlight any mix-ups in transferring sample tags from the received bags through the wet and dry lab processes. As of the date of this report, there have been no such instances.

ALS has its own internal QA/QC program, which is reported in the assay certificates sent to Lonmin, but it is not accounted for in determination of batch acceptance or failure. For geochemical and fire assays, ALS expects to achieve an accuracy percentage of ±10% (of the concentration), ±1 Detection Limit (DL) for duplicate analyses, in-house standards, and client-submitted standards when conducting routine geochemical analyses for gold and base metals. These limits apply at—or greater than—fifty times the limit of detection. For samples containing coarse gold, native silver, or copper, precision limits on duplicate analyses can exceed ±10% (of the concentration).

For ore grade analysis, ALS expects to achieve an accuracy percentage of ±5% (of the concentration), ±1 DL for duplicate analyses, in-house standards, and client-submitted standards. These limits apply at fifty times the limit of detection. As in the case of routine geochemical analyses, samples containing native silver or copper are less likely to meet the expected precision levels for ore grade analysis.

11.6 2016–2017 DRILL PROGRAMS

11.6.1 Borehole Core Sampling and Assay

For the 2015–2017 drilling campaign, core that was sampled outside the confines of the mineral envelope was sampled at 5 ft (1.5 m) lengths; core sampled within the mineral envelopes and the immediately surrounding 30 ft (10 m) was sampled at 3.3 ft (1 m) lengths (Baker and Hoffman, 2017). Exceptions occurred where a geological unit was narrow (i.e., less than 3.3 ft), or to make up the length between a sample above and a geological contact. The minimum sample length is 30 cm (6 in).

Core was cut in half with a water-cooled saw. One half was placed into a sample bag with a bar-coded sample tag, and the other half was stored at the Lonmin office at 129 Fielding Road, Lively, Ontario, as a representative sample or for possible re-sampling.

Samples were weighed, placed in plastic bags, grouped together in rice bags for transport, and delivered to ALS Chemex by Lonmin personnel.

11.6.2 Density Data

All diamond drill core from the 2015–2017 drilling campaign sourced from within the expected intersection of the mineral envelope was subject to SG measurement. This was completed by measuring dry and submerged sample weights. Each sample was allowed to fully dry after being cut, was weighed on top of the balance, placed in a mesh basket suspended from a free-hanging hook below the balance, and weighed submerged in water. The water was kept at approximately 20°C using a heater/agitator. Density was calculated using the following formula, where 0.998 is a factor to account for the lower density of water at 20°C:

$$\text{Density} = \text{Dry weight} / (\text{Dry weight} - \text{Wet weight}) * 0.998$$

The balance was checked with reference weights, and a reference rock sample had SG determined prior to each day of SG determinations to produce high quality data.

Several 2015–2017 drilling campaign samples within the mineral envelope do not have SG measurements because where they were sourced from was not—at the time—expected lie within the mineral envelope. For these samples and all historical data, a regression was applied in Vale's MEBS database to calculate an estimated density. The Alcock formula, as follows, was used to calculate density based on assay results for all samples where Cu, Ni, and S assays are available:

$$\text{Density} = 100 / (100 / 2.88 + 0.0166*\%\text{Cu} - 0.1077*\%\text{Ni} - 0.328*\%\text{S})$$

For samples drilled before 1968, for which only Cu and Ni are available, the following formula was used:

Density = 2.80 + 0.02*%Cu + 0.20*%Ni

These formulae were developed for semi-massive to massive contact Ni-Cu sulphide deposits. They are known to underestimate the density of most felsic rocks and overestimate the density of most mafic rock types outside the SIC The lowest possible values from the Alcock and pre-1968 formulae are 2.88 and 2.80 g/cc, respectively—too high for the felsic FW lithologies. The formulae also underestimate the density of mafic rocks. As a result, there is little correlation between calculated values up to 3.00 g/cc and measured values. Sulphide contributes more significantly to the density of samples.

There are 1,805 measured density values that were merged into the borehole database, a subset of which lie within the mineral envelope. A new density field for use in block modelling was added to the borehole dataset, where measured density overrides calculated density, if available. As the bulk of the 9400 Zone is hosted by mafic rocks, the density used in this grade model and the resulting tonnage could be biased lower. As highlighted in the exploratory data analysis, a revised density calculation could be used in the future to improve this bias. This was not applied in the current resource estimation as the revised density calculation was developed after the bulk of this mineral resource estimation was complete.

A density of 3.01 g/cc has been assigned to the Olivine Diabase dykes, which is both the median and mean value of 57 measured values for the unit in the 9400 Zone area. The stopes are all assumed to be filled with rock fill because rock fill was recovered each time a stope was encountered. The stopes are assumed to be approximately two-thirds rock fill and one-third void space, and were assigned a density of 2.00 g/cc. Other mine workings (i.e., air raises, drifts, and escape ways) are assumed to be void space and have been assigned a density of 0.

11.6.3 Data Management

All borehole data from surface and underground drilling are stored in Vale's MEBS database. In addition, all boreholes drilled in the 2015–2017 campaign exist as MS Excel® files exported from MEBS to serve as back-up records.

Data—including core photos, geotechnical logs, and measured density data—are uploaded into MEBS, and copies from the 2015–2017 campaign are retained. Copies from all previous drilling campaigns since the inception of the joint venture have been obtained from Vale.

MEBS contains data from holes logged directly in the system and holes that existed previous to the database, dating back to 1901 on the Denison property. Old boreholes often have extremely short interval descriptions, if any. The borehole database used in this resource estimation was downloaded by Alexander (Sandy) Gibson of Vale on 27 April 2017 in Datamine Studio 3, using Vale Ontario Operation's scripts, which export data directly out of MEBS. The measured density data was manually merged with this dataset using the sample identification number as the key field.

11.6.4 2015–2017 Borehole Assays and QA/QC

During the 2015–2017 Denison drilling program, diamond drill core was transported from the Denison project site to the Lonmin core shack—a distance of about 30 km—by either company personnel or by the drill contractor. The core was inspected for continuity and the correct markings of depth; tagged; and then logged; sample intervals were marked by Lonmin geologists.

Cores were halved using a water-cooled diamond saw that was cleaned regularly to avoid sample-to-sample contamination. One half of each core was submitted to the lead laboratory—ALS Minerals in Sudbury—by Loncan staff for analysis; the other half was retained on outdoor, roofed core racks at the Loncan office at 129 Fielding Road in Lively, Ontario, as a representative sample or for possible re-sampling. Prior to dispatch to the sample analysis laboratory, each individual sample was weighed.

At Denison, two variable styles of mineralisation are typically juxtaposed or located in proximity to each other, namely high sulphide (contact style mineralisation) and low sulphide (FW style mineralisation). Occasionally, stringer type mineralisation is also developed. Drill core sampling is guided by lithology, alteration, or visible mineralization; however, due to the nature of the low-sulphide mineralisation at Denison, sampling is routinely extended over the entire expected mineralised interval, and extended to a wider buffer zone on either side. The sampling interval is continuous, with no gaps left where mineralisation is perceived to be absent.

The maximum sample length is 5 ft (1.5 m) and the minimum sample length is 30 cm (6 in).

To make the data compatible with Vale's MEBS program, marking, logging, and sampling are recorded in imperial rather than metric units. Every reasonable effort is made so that that the sample sent to the laboratory is representative of the entire section of core; however, due to nugget effects and the heterogeneity that is common with this type of PGE mineralization, it is not guaranteed that an assay could be repeated. The half-core samples selected for assay are un-orientated. All samples are sealed (i.e., stapled) in individual, labelled plastic bags with a unique sample tag.

Blind CRMs—referred to as standards—and field silica blanks were included in sample runs, and submitted on the basis that the minimum oven batch size at the lab is in the range of 20 to 24 samples. Standard procedure is to include in each batch one CRM, one blank sample, and one sample duplicate, which is two quarter cores from the same half core sample. The CRM was inserted randomly into the batch, while the blank sample was placed in sequence immediately after where the highest PGM grade was expected. The position of the sample duplicate was random. The sample book used to track the samples was in the standard Vale format, using their numbering system to allow for easy integration of assay results into their borehole database.

In previous drilling campaigns, standards and blanks were inserted at a ratio of one blank and two standard per 100 samples. Crusher rejects were duplicated at the laboratory at a rate of three per 100 samples (see Appendix A).

Samples are delivered by Lonmin personnel exclusively to ALS Minerals in Kelly Lake Road, Sudbury, Ontario. They are then booked in their Laboratory Information Management System (LIMS) and batched before entering their prep lab. Both blank and standard sample insertion also serve to highlight any mix-ups in transferring sample tags from the received bags through the wet and dry lab processes. A further check is a comparison of sample mass delivered versus sample mass recorded as being received at the lab.

The ALS Canadian analytical laboratories are accredited by the SCC for specific tests listed in ALS's Scopes of Accreditation, which conforms with CAN-P-1579: Requirements for the Accreditation of Mineral Analysis Testing Laboratories; and CAN-P-4E ISO/IEC 17025: General Requirements for the Competence of Testing and Calibration Laboratories.

Becoming ISO-accredited involves detailed, on-site audits to evaluate ALS's quality management system and verify the technical competence of methods and personnel. This technical verification includes the requirement for successful participation in inter-laboratory proficiency testing programs and full method validation.

At ALS Minerals, received samples are checked against requisition documents prior to being dried, weighed, and then crushed to 70% passing -2mm. They are then Boyd rotary split to 250 g, and this is pulverized and split to better than 85% passing 75 microns. The 250-g splits are then transported by ALS Minerals to their analytical facilities in Vancouver, British Columbia via courier. Coarse and pulp rejects are retained at the Sudbury facility for a minimum period of six months; however, these are routinely collected by Lonmin personnel for storage at the Lonmin office facility.

For routine analysis at ALS, samples undergo the proprietary PGM-ICP23 process, which involves fire assay with standard lead collection of a 30-g aliquot for Pt, Pd, and Au. This is followed by a combination of ICP-MS and ICP-AES to finish the samples.

Samples that exceed 10 g/t on any individual PGE are also run through the PGM-ICP27 process, which recalibrates the ICP-AES finish to accurately report values up to 100 g/t for the PGE. Samples are also analyzed for 33 trace elements and base metals (including Ni, Cu, Co, and Cr) using a four-acid (HNO3-HClO4-HF and HCl) near total digestion, and a combination of ICP-MS and ICP-AES (ME-ICP61 process). ICP-MS over-limits for the ME-ICP61 process are reanalyzed using HF-HNO3-HClO4 acid digestion, HCl leach, and ICP-AES (ME-OG62 process).

In the event of visibly higher-grade mineralization, the preference is to analyze sample groups by submitting the samples directly for the analytical methods described for over-limits, with a specific sample tag prefix. These analytical methods, also referred to as High Grade/Ores Methods, are comprised of HF-HNO3-HClO4 acid digestion, HCl leach, and ICP-AES.

In addition to High Grade/Ores Methods, sulphur is analyzed using Total Sulphur by LECO to accommodate the anticipated higher sulphur levels. ALS Minerals provides assay results to Loncan's Senior Manager, Exploration and Project Geologist via e-mail in MS Excel® format.

The CRMs used on the Denison program from 2014 are PGM standards PGMS-24 and PGMS-25. Subsequently, in 2016, two additional standards—PGMS-27 and PGMS-28— were introduced to the QA/QC process due to the exhaustion of stocks of PGMS-24.

All CRMs were obtained from CDN Laboratories in Vancouver—the first two are sourced from Stillwater (specifically the J-M Reef), both are low Ni-Cu, one is low in 3E (PGMS-25; 2.7 g/t), and one is moderate in 3E (PGMS-24; 6.7 g/t). The second two were made available in 2016 and have a different provenance;

PGMS-27 is sourced from the skarn-related Serra Pelada Au-PGE deposit in Brazil, with a moderate 3E grade of 8.09 g/t 3E; and PGMS-28 comes from the low-grade (3.45 g/t 3E) Platreef from the central section of the Bushveld Complex's Northern Limb. In PGMS-27, Au forms 65% of the precious metal assemblage, while it forms only 5% in PGMS-28.

Standards are supplied in batches of one hundred 50-g envelopes via courier from Vancouver. No separate Ni-Cu standards are used. Standards are inserted randomly in the sample order. Blank samples of quartz sand are also inserted in the sample order immediately after an expected high-grade PGE/Ni-Cu sample.

Upon receipt of the assay data from ALS, the samples representing CRMs, blanks, and sample duplicates are highlighted and compiled manually for inspection. Assay values are denominated in Avoirdupois ounces per short ton, and these are converted directly into metric grams per tonne using a conversion factor of 34.28657. The values are then imported into a separate MS Excel® monitoring sheet and plotted graphically.

The QA/QC tolerances and hurdles for the Project are based on the means and standard deviations of the round robin laboratory data for the individual PGMS standards. A batch failure is considered to be when any of the individual 3E assays for a standard sample in the batch exceeds the mean ±3SD threshold, or when more than one sample in a batch exceeds the mean ±2SD threshold on any of the 3E.

The MS Excel® monitoring sheet is conditionally formatted on a True or False basis for each individual precious metal, such that a failure to meet the 3SD criterion is flagged as a False entry and highlighted. The assay values are plotted together on a time-ordered scatter plot graph for each individual envelope with the round robin laboratory data—on which the accepted mean and variance values for the standard are based. The graph (as shown in Figure 11-2) displays an envelope bounded by the *mean + 3SD* and *mean – 3SD* thresholds, and failures lie outside of that envelope.

The graphs serve to show any overall and between-batch bias for each of the elements in each of the standards, as well as the overall precision. As of the date of this report, there is a small positive variance for each of the individual 3E relative to the round-robin means, but no discernable trend with time. The precision in the ALS data is generally better than that of many of the round-robin labs used to compile the standards.

A batch failure would necessitate repeat assay of the entire batch, from coarse reject stage, with new control samples. Since inception of the QA/QC protocol, there have been three element failures related to PGMS-24 (Au for all three) and three related to PGMS-25 (Au for one, Pt for two). The specific instrumentation or fusion runs containing these samples were identified and re-run for assay. In all cases, there were no significant differences between the original and repeat assays. The problem was attributed to difficulty in achieving fusion of the original CRM samples.

With the later introduction of the PGMS-27 and PGMS-28 CRMs, it became apparent that the precision on individual elements in PGMS-27 was low. While there have been no recorded failures on the mean ±3SD criterion, and fewer than ten reporting outside the mean +/- 2SD lines, the spread of data is significant. For PGMS-28, problems have been experienced with the Au assay which has a mean value of 0.193g/t in the round robin data. Given the proximity to the detection limit in a routine commercial lab fire assay, it is considered likely that the population for this element would be closer to lognormal rather than normal and that the upper and lower cut-offs that are selected based on a normal distribution are not appropriate. During the 2016/17 drilling program there were four failures on Au only in PGMS-28, two on the high side and two on the low side. In the first instance the surrounding ten samples in the containing batch were rerun without significant variance. Subsequently, the three other failed batches were examined and, as there were no significant values in the surrounding ten samples, no further action was taken beyond flagging of the data. There have been no failures on either Pt or Pd for this standard.

For blank sample values, failure is more subjective, and a failure ceiling value has not been set – early on in the program very occasionally there were instances of blank values returning up to 0.24 g/t 3E where it is likely there had been carry over from the previous high grade sample in the prep stage. This issue became noticeably more apparent during 2016 with the inception of the Denison 109FW metallurgical drilling program where one third HQ core samples were being submitted instead of the normal one half core NQ. In these instances a 3 kg plus high grade sample was being crushed and followed by a 100 100-g quartz blank, with the effect that any carryover had a disproportionately high effect and was being magnified in the reported blank grade. Sporadic values of up to 0.695 g/t 3E were recorded in blank samples because of this. In these instances, the lab was notified of this issue—in all cases, the mass carryover from one sample to the next was within the contractually acceptable tolerances, which are set at a percentage.

From the beginning of 2016 until end of April 2017, a total of 8,786 samples were submitted for assay. These included 146 samples of standard PGMS-24, 176 of PGMS-25, 33 of PGMS-27, and 32 of PGMS-28, as well as 390 blank samples. QA/QC coverage is therefore 8.8%, which is close to the planned coverage of one standard plus one blank per nominal oven batch of 24 samples.

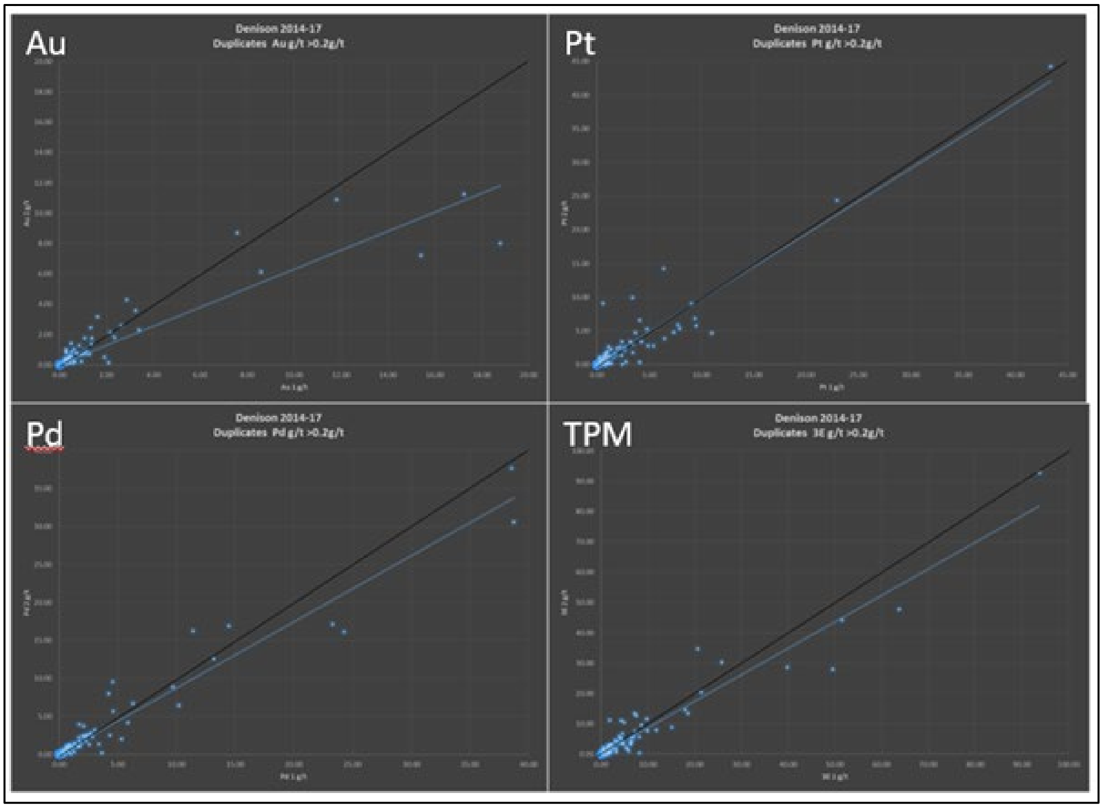
ALS has its own internal QA/QC program, which is reported in the assay certificates sent to Lonmin, but it is not accounted for in determination of batch acceptance or failure. For geochemical and fire assays, ALS expects to achieve an accuracy percentage of ±10% (of the concentration), ±1 Detection Limit (DL) for duplicate analyses, in-house standards, and client-submitted standards when conducting routine geochemical analyses for gold and base metals. These limits apply at—or greater than—fifty times the limit of detection. For samples containing coarse gold, native silver, or copper, precision limits on duplicate analyses can exceed ±10% (of the concentration).

For ore grade analysis, ALS expects to achieve an accuracy percentage of ±5% (of the concentration), ±1 DL for duplicate analyses, in-house standards, and client-submitted standards. These limits apply at fifty times the limit of detection. As in the case of routine geochemical analyses, samples containing native silver or copper are less likely to meet the expected precision levels for ore grade analysis.

Duplicate samples were submitted for the bulk of the 2015–2017 drilling campaign as a means of investigating the precision at ALS. As duplicates were submitted at the target rate of one per 24 samples spread evenly throughout each borehole, the bulk of the duplicated samples were low grade, making the precision analysis difficult. For this reason, duplicate sampling was discontinued toward the end of the drilling campaign.

Duplicate samples are plotted in Figure 11-3, with the first sample on the X axis and the second sample on the Y axis for all of the precious metals, as well as the combined TPM grade. Black lines represent 100% precision and blue lines represent linear regression.

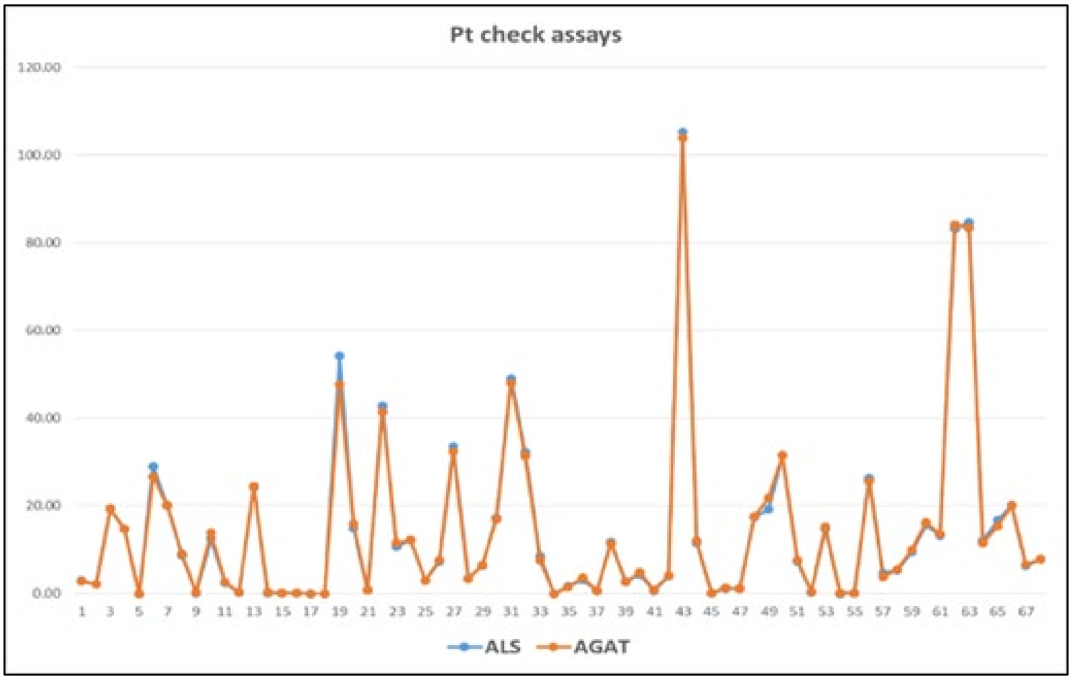
Figure 11-3: Duplicate Assay Analyses for Au, Pt, Pd, and TPM (Baker and Hoffman, 2017)



The linear regressions for Pd, TPM, and Pt are very close to the 100% precision line. The linear regression for Au shows the greatest deviation from 100% precision, due to five of the six Au samples over 6 g/t having higher grades in the first sample than the second, skewing the regression line downward.

The use of a third-party laboratory for routine check assays was investigated during 2016 as a means to investigate the slight—but consistent—positive bias seen in ALS CRM assays. The aim was to match ALS's methodology as closely as possible. AGAT Labs were selected, and a trial run of 68 variably high-grade channel sample pulps from a prospect on Denison was renumbered and submitted for analysis. The results returned from AGAT, as plotted in Figure 11-4, showed exceptional precision with respect to the corresponding ALS analyses, without exception.

Figure 11-4: Pt Check Assays on A Batch of 68 Pulps from AGAT (Orange) Overlain on Original ALS Assays (Blue) (Baker and Hoffman, 2017)



Unfortunately, AGAT closed their precious metal lab shortly after completion of this work and no replacement third-party lab has been chosen.

11.7 QUALIFIED PERSON'S COMMENTS

It is the Author's opinion, based on a review of all possible information, that the sample preparation, analyses, and security used on the Project meet acceptable industry standards. The drill data can therefore be used for geological and resource modelling, and estimation of Indicated and Inferred mineral resource.

12 DATA VERIFICATION

This section summarises the data verification procedures that were carried out and documented by the Author for this report.

As part of the verification process, the Author reviewed all geological data and databases, as well as past in-house technical reports.

12.1 SOURCES OF EXISTING DATA

To date, Magna has completed 68 surface diamond drill holes between Q4 2022 (November) and Q3 2023 for a total of 12,180 m. Results of the Magna drilling have not yet been considered in the geological interpretation, and assay results have not been incorporated into the mineral resource estimate. The 2022-2023 drill data has not been reviewed by the Author. The effective date of the mineral resource remains 19 August 2022.

All previous drilling is described in Section 6: History and Section 10: Drilling. The Author assumes that the sample preparation, analyses, and security for drilling completed by other issuers prior to the effective date of this report has been reviewed and validated by previous authors of internal resource estimates, including WSP (2020) and SRK (2020). Armitage believes that sample preparation, analysis, and security by previous operators, as described in this report, were completed in a manner consistent with industry standard sampling techniques at the time.

12.2 VERIFICATION METHODS

Armitage conducted an independent verification of the assay data in the drill sample database. Approximately 10–20% of the digital assay records were randomly selected and checked against the available laboratory assay certificate reports by Armitage.

Only assay certificates were available for drilling completed by Loncan from 2014 to 2017. Assay certificates for drilling by Vale prior to 2014 were not available.

Armitage reviewed the assay database for errors, including overlaps and gaps in intervals, and typographical errors in assay values. In general, the database was in good shape and no adjustments were required to be made to the assay values contained in the assay database.

Verifications were also carried out for drill hole locations, down hole surveys, lithology, SG, trench data, and topography information. Minor errors were noted and corrected during the validation process but had no material impact on the 2022 MRE presented in this report. The database is of sufficient quality to be used for the current MRE.

In addition, the Authors conducted a site visit to better evaluate the veracity of the data.

The property is a past producing mine and is currently at an advanced stage of exploration. The project has had numerous studies completed, and has had numerous past authors (i.e., Lonmin, Loncan, WSP, and SRK) complete site visits, data verification programs, internal mineral resource estimates, and mineral resource estimate reviews of various parts of the deposit. The Project has seen past production (open pit and underground). As such, the Author did not deem it necessary to collect check samples.

12.3 MAY 2022 SITE INSPECTION AND DATA VERIFICATION

Armitage inspected the Property on 25 May 2022, accompanied by Jason Jessup, CEO and Director of Magna; David King of King Geoscience, Technical Advisor and QP for Magna; and Dave Smith, Senior Geologist for Wallbridge Mining Company Ltd. Armitage completed a tour of the historic mine site, including the area of the shafts and raises, previous open pit, and waste dump. The Author visited a number of outcrops to review the geology and various styles of mineralization, rock sample and channel sample locations, and recent and historical drill sites.

On 26 May 2022, the Author was able to visit the Project's core storage facility in Sudbury (i.e., Wallbridge core storage facility), accompanied by David King and Dave Smith. Armitage examined a number of selected mineralized core intervals from recent diamond drill holes from the Project. Armitage examined assay certificates, and assays were examined against the drill core mineralized zones.

All core boxes were well labelled and properly stored in core racks outside, with a number of significant drill intercepts stored on core racks inside. Sample numbers for recent drill holes were written on each core, and it was possible to validate sample intervals and confirm the presence of mineralization in witness half-core samples from the mineralized zones.

At the time of the visit, there was no active exploration or mining activities on the property. As of the date of this report, Magna has not completed any further exploration on the property.

As a result of the site visit, the Author was able to become familiar with conditions on the property; was able to observe and gain an understanding of the geology and various styles of mineralization; was able to verify the work done; and, on that basis, can review and recommend to Magna an appropriate exploration or development program.

The Author considers the site visit current, per Section 6.2 of NI 43-101CP. To the Author's knowledge, there is no new material scientific or technical information about the property recorded since that personal inspection. This report contains all material information about the property as of the date of its publication.

12.4 CONCLUSION

All geological data has been reviewed and verified by the Author as being accurate, and all geologic information was reviewed and confirmed, to the extent possible. There were no errors or issues identified with the database. Based on a review of all possible information, the Author is of the opinion that the database is of sufficient quality to be used for the current Indicated and Inferred MRE.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The deposit is made up of the remnants of the historic mine operations of the Crean Hill Mine, and extensions into the FW adjacent to the historic mining. The geometallurgical types typical in the Sudbury Basin are represented in the deposit. Approximately 30% of the mineralization consists of FW and LSHPM material identified in the 109 FW adjoining the historic deposits.

The 109 FW material has been the subject of numerous evaluations, which are discussed in this section. The contact ore, which represents approximately 70% of the mineralization, is contained in extensions and remnants of the Ni mines and represents contact type material. Contact mineralization is known to respond to flotation, and recovery of Ni is dependent on pyrrhotite content. The contact mineralization of the resource has not yet been tested at the time of publication of this report but will be tested in subsequent project phases.

Vale performed mineralogy and metallurgical test work on 109 FW material in 2010 and 2011. Lonmin completed mineralogy and additional metallurgical testing on the additional material in 2016 and 2017.

The following presentations, reports, and studies were referenced to source information for this section of the report:

1. 1-Jun-2010 – F.Ford et.al. - Vale Internal Report – Process Mineralogy of PGMs from Denison 109FW Zone Sample 1244
2. 17-Jun-2010 – X. Manqui et.al. - Vale Internal Report – Mineralogy and Metallurgy of Denison PGM Ore
3. 31-May-2010 – A. Lee et. al. - Vale Internal Report – Phase 2 Metallurgical Evaluation of Denison PGM
4. 11-Jul-2012 – A. Lee et. al. - Vale Internal Report – Denison 109 Zone MinMet 2011
5. 11-Jul 2016 – L. Capri - Report 2016-03 - Precious Metal Deportment in Six Composite Samples from the Denison 9400 FW Zone
6. 26-Feb-2018 – A. Kelly - Blue Coast Research PJ5219 - Denison Pre-Feasibility Study
7. 29-Oct-2020 – A. Kelly - Blue Coast PJ5313 - Denison Project – Gravity Recovery and Flotation Optimization
8. 7-Apr-2022 – C, Gould et. al. – Vale - Technical Report Summary, Sudbury Property, Ontario Operations, Canada
9. 14-Dec-2022 – A. Armitage - SGS – Technical Report on the Mineral Resource Estimate for the Denison Deposit, Denison Project, Sudbury, Ontario Canada

13.2 MINERALOGY

There have been two mineralogical evaluations completed for the 109 FW zone: in 2007, Vale conducted an internal mineralogy evaluation on a sample from the upper part of the FW; and in 2016, Lonmin had six samples from the 9400 Zone (three from the upper zone and three from the deeper zone) analysed by Cabri Consulting Inc. The results are shown in the table presented in Figure 13-1.

Figure 13-1: Summary of Modal Mineralogy for FW Zones (Source: XPS, 2023)

	2007 Sample - 109 Footwall	2016 Samples - 109 Footwall 9400 Zone	
	100127_Lonmin_1206-2	Avg. Upper Zone	Avg. Lower Zone
Pentlandite	1.1	1.1	1.6
Chalcopyrite	1.2	1.7	7.7
Pyrrhotite	2.9	1.7	11.1
Pyrite	0.1	0.1	0.0
Millerite	0		
Cubanite	0.01		
Bornite	0		
Sphalerite	0.03	0.1	0.1
Galena	0.001		
Gerdorffite	0.001	0.2	0.1
Niccolite/Maucherite	0	0.0	0.0
Ni Marcasite		0.3	0.0
Total Sulphides	**5.5**	**5.1**	**20.7**
Olivine	0.01		
Orthopyroxene	0.01		
Clinopyroxene	1.77	0.58	0.20
Anphibole	37.8	29.94	25.18
Epidote	2.68	5.97	4.29
Chlorite	18.51	17.44	13.46
Biotite/Micas	4.23	5.49	5.25
Talc	0		
Serpentine	0		
Quartz	8.07	14.18	13.90
Plagioclase	16.1	15.94	12.72
Kspar	0.37		
Titanite		1.18	0.69
Total Silicate	**90.8**	**90.7**	**75.7**
Apatite	0.63	0.52	0.46
Magnetite	0		
Hemitite	0.01		
Ilenite	1.71	1.60	1.48
Calcite	0.84	1.89	1.43
Slag Glass	0.16		
Other		0.17	0.23
Total Phosphate, Oxide, Carb, other	**3.75**	**4.18**	**3.60**
TOTAL	100	100	100
Not Reported			

The 2007 samples and the 2016 samples of the upper zone were consistent in terms of mineralogy. The 2007 sample of the 109 FW contained approximately 5.5% sulphides; 90.8% silicates; and 3.75% Fe-oxides, carbonates, and other minerals. The three 2016 upper zone samples from the 9400 Zone contained an average of 5.1% sulphides; 90.7% silicates; and 3.66% Fe-oxides, carbonates, and other minerals. The three 2016 lower zone samples from the 9400 Zone were higher in sulphides, averaging 21% (mostly due to increased pyrrhotite); they therefore had correspondingly lower silicates, at 75%; and Fe-oxides, carbonates, and other minerals, at 3%.

Ni deportment was conducted on the 9400 Zone samples and results varied. The two economically recoverable minerals of pentlandite and gersdorffite contained 71% and 9% of the Ni, respectively. Pyrrhotite and marcasite contained 5% of the Ni, and silicates—primarily chlorites—contained 13% of the Ni. The Ni deportment will limit maximum Ni recovery from this material to 80%. There was negligible millerite—which would negatively impact the Cu-Ni separation efficiency—identified in any of the samples.

Liberation of the recoverable minerals in the 2016 material was good. The material was ground to an 80% passing size (i.e., P80) of between 75 and 100 µm, and the results were 78% liberation for pentlandite, 83% liberation for chalcopyrite, and 85% liberation for gersdorffite. These results meet the criteria for expected liberation required for flotation. Precious metal deportment for Pt, Pd, and Au was also conducted on the samples from both the 109 FW and the 9400 Zone.

Pt was identified in both studies; most of the Pt (i.e., >80%) occurred as part of the mineral sperrylite (PtAs2), and a small amount occurred as part of moncheite (Pt[Te,Bi]2). The 2016 report also identified up to 5% of the Pt to occur as part of gersdorffite (NiAsS), as a solid solution.

Pd was identified as occurring in the mineral michenerite ([Pd,Pt]BiTe). In the 109 FW Zone, Pd was almost entirely (i.e., 96%) associated with this mineral, whereas in the 9400 Zone, some samples carried up to 35% of the Pd in in gersdorffite (NiAsS), as a solid solution.

Au-bearing minerals were not commonly found and almost exclusively occurred as part of Electrum (Au,Ag) and native gold (Au>80%).

Precious metal bearing minerals were described as fine (in terms of size) in the 2016 evaluation and had a P_{80} of 23 µm.

13.3 HARDNESS TESTING

Hardness data is required to determine grinding requirements for achieving both the throughput and P_{80} product size necessary to achieve the metallurgical results. Hardness measurements were conducted for the 109 FW Zone in 2007, and additional samples were measured in 2016.

In 2007, results for the 109 FW Zone were reported for crusher, semi-autogenous (SAG) mill, rod mill, and ball mill parameters. The crusher work index was reported as 11.7 kWh/t, which is considered very hard.

For determining energy requirements for circuits consisting of SAG mills, the JKSimMet© A x b (unitless parameters derived from the JK drop test) is primarily used. The A x b value of 25 was determined, which indicates that the rock is competent and rated as very hard relative to other materials.

Bond work indexes are used for determining power requirements for rod and ball mills. The 2007 testing resulted in a Bond rod mill work index (RWi) of 18 kWh/t, and a Bond ball mill work index (BWi) of 16.2 kWh/t, which are both rated as hard compared to other materials.

Additional drill core from the 109 FW was tested in 2016. The sample was divided into an upper sample representing material from above 100 m in depth, and a lower sample representing material from below 100 m in depth. The RWi, BWi (at a closing size of 75 µm), and abrasion index (Ai) were determined for both of these samples. The upper sample was slightly harder than the lower sample, with a RWi of 19.1 kWh/t, a BWi of 19 kWh/t, and an Ai or 0.349. The lower sample—containing higher sulphides—was slightly softer, with a RWi of 17.4 kWh/t, a BWi of 16.6 kWh/t, and an Ai of 0.327.

The measurements indicate that the 109 FW material samples were all similar and rated very hard.

13.4 METALLURGICAL EVALUATIONS

Two metallurgical evaluations were conducted: Vale evaluated the Denison 109 FW in 2010, and Blue Coast performed an evaluation in 2018.

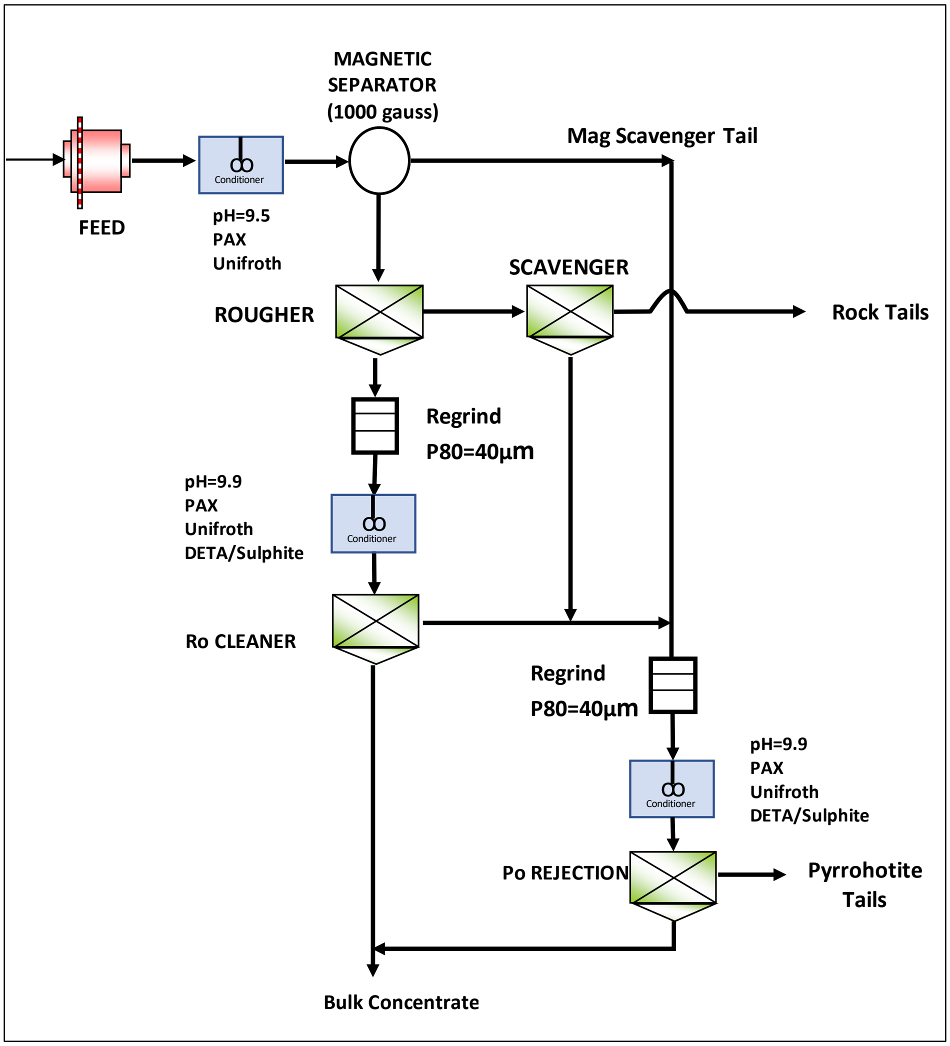
13.4.1 Evaluation of Denison 109 Footwall by Vale

The earliest test work on 109 FW was conducted by Vale. A series of tests were conducted on samples labelled 1206-1,1206-2, and 1206-3, representing the upper, middle, and lower section of drillhole LMO320. Based on promising Pd, Pt, and Au recoveries, a second phase of testing was initiated.

The feed sample for the second phase was selected from two intervals of two holes, and labelled as Denison 1244, to be representative of the Denison 109 FW drillhole LMO320. The samples averaged 0.14% Cu, 0.15% Ni, 0.73% S, 2.44 g/t Pd, 5.15 g/t Pt, and 1.42 g/t Au.

The response of the material was evaluated based on the Clarabelle Full Circuit Simulation (FCS), which was the standard test in 2010. This process is presented in Figure 13-2.

Figure 13-2: Vale Full Circuit Simulation (pre-2015) (Source: XPS, 2023)



The program first evaluated a range of blends of Denison to Clarabelle feed, as follows:

- 100% Denison feed
- 50% Denison feed to 50% Clarabelle feed
- 25% Denison feed to 75% Clarabelle feed
- 100% Clarabelle feed

The results from the 100% Denison feed are shown in the table presented in Figure 13-3.

Figure 13-3: Results of 100% Denison 109 FW using Vale FCS Flowsheet (Source: XPS, 2023)

Stream		Assay					Distribution (%)				
	Wt %	Cu (%)	Ni (%)	Pd g/t	Pt g/t	Au g/t	Cu	Ni	Pd	Pt	Au
Feed	100	0.21	0.22	2.38	4.30	1.31	100.0	100.0	100.0	100.0	100.0
Conc	2.4	6.85	4.65	43.00	84.90	19.40	79.6	51.6	44.1	48.2	36.6
Tails	97.6	0.04	0.11	1.36	2.28	0.85	20.4	48.4	55.9	51.8	63.7

In this series of testing, the distribution was approximately 44% Pd, 48% Pt, and 36% Au. Due to the low Cu+Ni grade of this sample, the concentrate grade produced was only 11.5% Cu+Ni. This is below the Clarabelle concentrate grade target of 20% Cu+Ni. Blending with the higher Cu+Ni Clarabelle feed resulted in targeted concentrate grades, at similar Pd, Pt, and Au recoveries.

The program tested a separate flowsheet with a finer grind of 64 µm and achieved higher Pd, Pt, and Au recoveries. The results were 62% Pd, 67% Pt, and 57% Au to a bulk concentrate of 9% Cu+Ni. It was not clear from the report how much of the added recovery was due to the finer grind versus the lower concentrate grade. To achieve this lower grade, the material would have to be processed on its own. When processed as a blend with low Pd, Pt, and Au value contact ores, existing processing plants will target +20% Cu+Ni grades, which would negatively impact the precious metal recovery.

In about 2012, Clarabelle updated their circuit to a new flotation configuration referred to as the challenging ore recovery (CORe) flow sheet.

Based on the change to the circuit, additional tests were conducted. The sample for this phase was labelled 1335 and assayed 0.60% Cu, 0.34% Ni, 2.13% S, 2.26 g/t Pd, 3.06 g/t Pt, and 1.14 g/t Au.

To calibrate the recovery model for this material, Vale performed two duplicate flotation tests on samples ground at standard and coarse grind sizes. The Clarabelle process targets >20% Cu+Ni concentrate grades, which are readily achieved with more typical and higher-grade Cu and Ni contact ores from the basin. The 109 FW, with its low Cu and Ni values, requires significantly more upgrading to achieve the 20% Cu+Ni concentrate grade threshold. Achieving these higher grades negatively impacts metal recovery.

The standard ground test resulted in a 78% Ni recovery to 20% Cu+Ni grade, and the coarse ground test achieved only 71% recovery to 20% Cu+Ni grade. Similarly, based on the standard grind Pd, Pt, and Au, recoveries were reported at 72% Pt, 70% Pd, and 68% Au, for a mass pull of 4%. A 4% mass pull was required for a 20% Cu+Ni grade. Recoveries from the coarse grind were reported at 64% Pd, 72% Pt, and 68% Au for a mass pull of 4%.

The Vale testing was focused on evaluating Denison 109 FW material's behaviour through the Clarabelle processing plant using their established grind, residence times, and flowsheet. They demonstrated that the Pt, Pd, and Au recoveries increased with mass pull; this is related to the Cu+Ni grade in the feed, and inversely related to Cu+Ni grade in the concentrate. The earlier work with sample 1244 had a Cu+Ni feed of 0.29% and a lower weight pull of 1.2% to achieve a 20% Cu+Ni concentrate grade, and resulted in lower Pd, Pt, and Au recoveries. The later sample 1335, with a Cu+Ni feed of 0.94% Cu+Ni, required a 4.1% mass pull and achieved higher Pd, Pt, and Au recoveries.

13.4.2 Evaluation of Denison 109 Footwall by Blue Coast

Separation Using Flotation

Blue Coast conducted a metallurgical program on a composite prepared from Denison drill core at the direction of Micon International Ltd. The composite for the metallurgical testing was assayed in triplicate and averaged 0.755% Cu, 0.285% Ni, 2.29% S, 4.42 g/t Pd, 3.66 g/t Pt, and 1.67 g/t Au.

A series of tests were conducted to determine the effect of grind size on rougher flotation recovery of Pd, Pt, and Au. Based on the results presented in the table presented in Figure 13-4, a primary grind with a P80 of 60 µm was selected.

Figure 13-4: Grind Sensitivity of Denison FW (Source: Blue Coast PJ5219 Report P.10; Table 10)

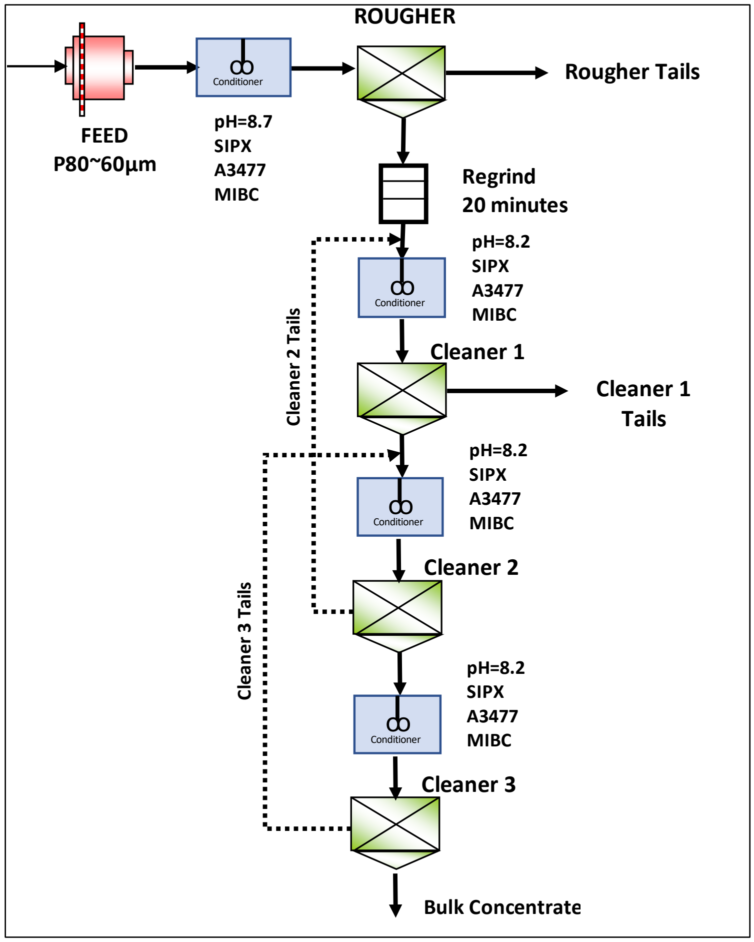
Test #	Grind P80	Grade					Rougher Flotation Recovery (%)				
	(µm)	Cu (%)	Ni (%)	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu	Ni	Pd	Pt	Au
F-4	184	5.08	1.77	16.86	20.93	6.22	97.3	83.8	78	85.8	62.4
F-5	121	5.83	1.8	36.29	21.65	7.48	97.4	83.3	80	88.7	67.5
F-6	82	5.86	1.8	16.78	21.68	8.78	98.1	84.4	82	88.6	72.8
F-7	69	5.55	1.71	17.56	21.19	9.28	98.3	84.4	83.4	90.1	76.3
F-8	56	5.21	1.68	32.3	23.49	8.94	98.8	85.7	86.7	89.7	78.4

A series of tests were next conducted to screen collectors for flotation. Based on the tests, a dual collector system of sodium isobutyl xanthate (SIBX) and sodium diisobutyl dithiophosphate (A3477) were selected because they provided overall better Pt recovery.

After finalizing the rougher conditions, cleaner tests were performed, and rougher concentrate regrinding was evaluated. The conclusion of the cleaner study was a recommendation of 30 minutes of regrinding required to achieve a Cu+Ni concentrate grade of greater than 12%.

Based on this development work the circuit presented in Figure 13-5 was finalized for lock cycle tests (LCT).

Figure 13-5: LCT Flowsheet (Source: Blue Coast PJ5219 Report P.26; Table 14)



A LCT is used to simulate the conditions of a plant where internal streams are circulated. There were two LCTs performed, each containing six cycles; stability was achieved in both tests (refer to Figure 13-6).

Figure 13-6: LCT Results (Source: Blue Coast PJ5219 Report P.27/29; Table 12/14)

LCT-1													
Stream		Assay						Distribution (%)					
	Wt %	Cu (%)	Ni (%)	S (%)	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu	Ni	S	Pd	Pt	Au
Cleaner 3 Conc	7.8	9.87	2.92	23.50	40.70	41.70	14.90	98.3	80.8	81.5	85.8	87.7	77.1
Cleaner 1 Tails	9.3	0.06	0.17	4.11	1.26	2.53	0.78	0.8	5.4	16.8	3.1	6.3	4.8
Rougher Tails	82.9	0.01	0.05	0.04	0.49	0.27	0.33	1.0	13.8	1.6	11.0	6.0	18.1
Feed	100	0.79	0.28	2.25	3.70	3.71	1.51	100	100	100	100	100	100
LCT-2													
Stream		Assay						Distribution (%)					
	Wt %	Cu (%)	Ni (%)	S (%)	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu	Ni	S	Pd	Pt	Au
Cleaner 3 Conc	5.5	13.60	3.76	29.40	57.80	56.50	19.70	97.3	75.1	73.0	82.4	85.4	73.0
Cleaner 1 Tails	10.9	0.10	0.26	4.99	1.96	2.49	0.90	1.4	10.4	24.6	5.6	7.5	24.6
Rougher Tails	83.6	0.01	0.05	0.06	0.55	0.31	0.37	1.3	14.6	2.4	12.0	7.1	2.4
Feed	100	0.77	0.27	2.21	3.85	3.63	1.49	100	100	100	100	100	100

For LCT-2, the mass pull was reduced to achieve a higher concentrate grade. The test resulted in a final concentrate with a grade of 17.36% Cu+Ni, with marginally lower Cu recovery at 97%, and lower Ni recovery at 75%. Precious metal recoveries were also marginally lower, with 82% of the Pd, 85% of the Pt, and 73% of the Au.

These tests demonstrate the sensitivity of this material to concentrate grade. The higher the concentrate grade targeted for this material, the lower the recoveries achieved.

Gravity Separation Combined with Flotation

During the 2016 pre-feasibility study by Blue Coast, four gravity separation tests were conducted to determine if Pd, Pt, and Au recoveries could be improved compared to the flotation-only process. After processing the ground material through the Knelson concentrator, the Knelson reject was subject to flotation using the standard conditions. The results are presented in the table presented in Figure 13-7.

Figure 13-7: Gravity/Flotation Test Results (Source: XPS, 2023)

			Grade					Distribution				
	Test #	Wt %	Cu %	Ni %	Pd g/t	Pt g/t	Au g/t	Cu	Ni	Pd	Pt	Au
Knelson Conc @ 57 µm Grind	*G-1*	*0.84*	*3.1*	*3.8*	*105.5*	*233.5*	*78.1*	*3.5*	*11.5*	*23.6*	*60.5*	*35.7*
Additional Rougher Conc.	*F-18*	*13.25*	*5.4*	*1.5*	*17.3*	*7.9*	*6.0*	*95.8*	*71.5*	*61.2*	*32.5*	*43.7*
Combined Grav/Flot Conc.		14.1	5.2	1.6	22.5	21.3	10.3	98.3	85.7	86.7	93.0	79.4
Flotation Only Rougher Conc.	F-8	14.1	5.5	1.7	23.5	32.3	8.9	98.8	85.7	86.7	89.7	78.4
Knelson Conc @ ~60 µm Grind	G-2	0.53	3.3	4.4	120.5	241.5	95.9	2.4	8.7	18.6	52.6	29.4
Additional Rougher Conc.	F-23	15.95	4.4	1.3	14.3	6.0	5.3	96.5	76.3	65.9	39.2	48.8
Combined Grav/Flot Conc.		16.5	4.4	1.4	17.7	13.6	8.2	98.9	85.0	84.5	91.7	78.3
Flotation Only Rougher Conc.	F-12	15.2	5.0	1.6	23.3	23.8	9.0	98.8	85.5	87.5	94.0	79.7
Knelson Conc @ ~100 µm Grind	*G-3*	*0.87*	*1.7*	*2.6*	*69.5*	*122.5*	*19.7*	*2.0*	*9.2*	*17.7*	*46.5*	*12.7*
Knelson Conc @ ~60 µm Grind		*0.89*	*3.9*	*3.7*	*45.8*	*44.7*	*14.4*	*4.8*	*13.1*	*11.9*	*17.2*	*9.5*
Additional Rougher Conc.	*F-24*	*15.32*	*4.4*	*1.1*	*12.8*	*4.3*	*4.9*	*92.1*	*68.5*	*57.5*	*28.7*	*55.9*
Combined Grav/Flot Conc.		17.1	4.2	1.3	17.4	12.4	6.2	98.9	90.7	87.0	92.4	78.0
Flotation Only Rougher Conc.	F-12	15.2	5.0	1.6	23.3	23.8	9.0	98.8	85.5	87.5	94.0	79.7
Knelson Conc @ ~60 µm Grind	G-4	0.95	3.5	3.9	106.0	216.0	61.2	4.4	13.6	24.8	61.6	32.3
Additional Cleaner 3 Conc.	F-28	7.13	9.7	2.6	33.6	14.3	11.0	96.5	68.7	59.1	30.7	43.6
Combined Grav/Flot Conc.		8.1	9.0	2.7	42.1	38.0	16.9	95.0	82.3	83.9	92.3	75.9
Flotation Only Cleaner 3 Conc.	F-20	7	10.5	3.3	45.9	38.2	14.2	97.4	81.9	81.6	85.5	70.2

The gravity combined with flotation resulted in very similar overall recoveries when compared to rougher flotation tests.

A decision was made in 2020 to advance the understanding of gravity separation through an Extended Gravity Recoverable Gold and PGE (EGRG+PGE) test. The test was conducted on the 2017 Denison 109 FW Master Composite.

A single EGRG+PGE test was conducted on a 10-kg sample of the Denison composite used in the 2017 study. An EGRG test involves the sequential processing of the same 10-kg feed sample at various grinds. The first pass through the Knelson is at a P80 of 850 µm, followed by a second pass after regrinding the first pass tails to a P80 of 250µm, and then a final pass after regrinding to a P80 of 75 µm. The results are shown in Figure 13-8.

Figure 13-8: Gravity Recoverable Gold and Precious Metal Test Results (Source: XPS, 2023)

Product	Grind Size	Mass		Assay			Distribution (%)		
	P80 (µm)	(g)	(%)	Pd (g/t)	Pt (g/t)	Au (g/t)	Pd	Pt	Au
Stage 1 Concentrate	850	99.3	0.51	126.3	408	59.2	16.4	57.1	16.6
Stage 2 Concentrate	250	87.1	0.45	77.4	88.8	18	8.8	10.9	4.4
Stage 3 Concentrate	75	101.5	0.52	78.4	63.5	24.8	10.4	9.1	7.1
Total Concentrate		**288**	**1.48**	**94.6**	**189.8**	**34.6**	**35.6**	**77.1**	**28.2**
Total Tailings		19139.4	98.5	2.57	0.85	1.32	64.4	22.9	71.8
Calculated Head		*19427.4*	*100.00*	*3.94*	*3.65*	*1.81*	*100.0*	*100.0*	*100.0*

The EGRG+PGE test work identified that Pt showed a very good gravity response, with 77% of the total Pt being potentially gravity recoverable. Gravity-recoverable Pd and gravity-recoverable Au values were lower, at 36% and 28%, respectively.

Nine additional gravity / flotation tests were conducted to simulate processing at the Redstone Mill in Timmins. The process followed is detailed in Figure 13-9 and the results obtained are presented in Figure 13-10.

Figure 13-9: Gravity Concentrator + Flotation Flowsheet (Source: Blue Coast PJ5313 Report P.26; Figure 14)

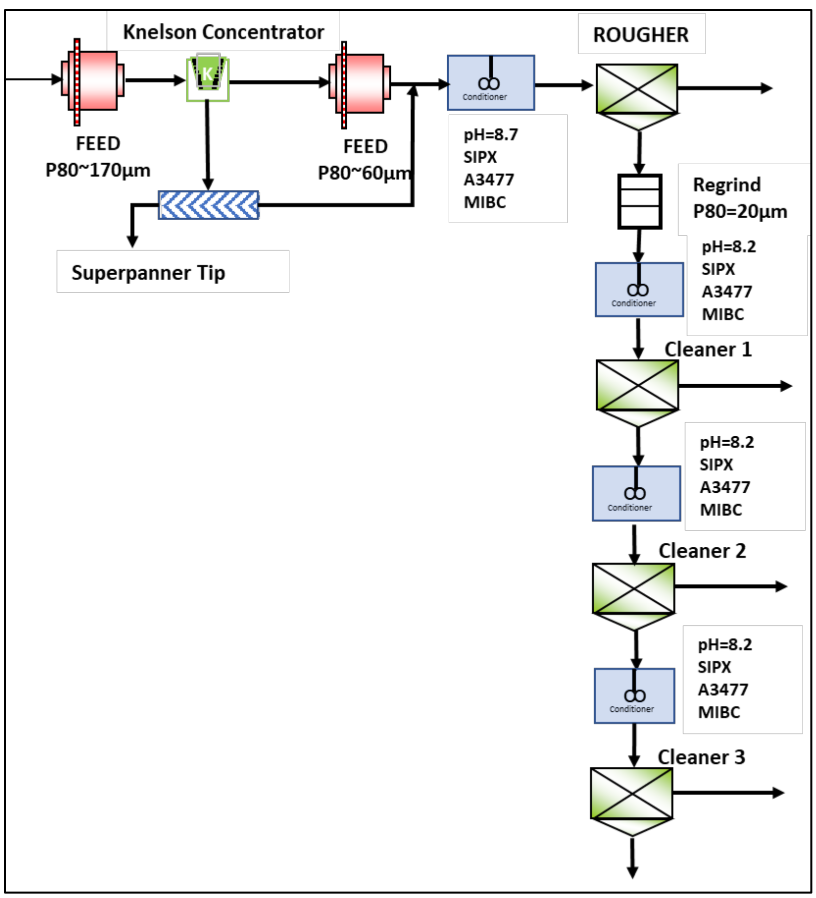


Figure 13-10: Open Circuit Gravity/Flotation Test Results (Source: XPS, 2023)

Test #	Knelson	Flotation	Regrind	Concentrate	Concentrate Grade					Concentrate Recovery (%)				
	P80 (µm)	P80 (µm)	(min)	Mass Pull (%)	Cu (%)	Ni (%)	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu	Ni	Pd	Pt	Au
F-1	175			0.02			365	1551	76			2.7	17.3	1.1
		70	10	4.08	11.25	3.28	42.2	24.4	13.0	63.4	49.8	51.8	45.3	31.8
F-2	175			0.03			851	2843	375			7.1	32.5	7.3
		69	20	6.57	10.90	3.32	40.7	20.8	14.3	95.2	76.8	76.8	54.1	63.6
F-3	175			0.03			940	2658	130			8.6	36.6	7.4
		69	22	6.51	11.39	3.25	42.3	19.3	14.6	95.1	75.0	73.0	49.8	61.7
F-4	175			0.01			821	5188	605			3.0	28.0	5.6
		68	22	5.99	10.56	3.04	40.1	19.7	12.9	85.6	68.1	70.1	50.9	57.2
F-5	175			0.18			342	1230	226			16.4	59.1	23.4
		75	22	6.73	11.35	3.25	37.4	18.3	13.5	94.8	76.0	66.2	32.5	51.7
F-6	67			0.15			280	907	211			10.7	46.4	17.4
		67	22	6.57	11.19	3.31	42.9	20.2	13.9	94.9	81.9	71.3	40.7	49.9
F-7	175			0.53			161	342	80			22.6	60.7	25.4
		68	22	6.49	11.57	3.18	35.7	13.5	13.0	94.6	70.8	61.9	29.6	51.3
F-8	67			0.19			354	945	321			17.9	61.9	33.8
		67	22	6.70	10.38	3.11	37.7	12.7	12.2	95.8	76.1	66.8	29.1	45.0
F-9	67			0.21			289	812	207			15.6	58.9	26.3
		67	23	6.71	9.73	2.75	37.3	13.0	11.4	91.3	71.7	65.8	30.7	47.3

Using the EGRG+PGE test results, a modelling exercise was conducted by FLSmidth Knelson. The modelling demonstrated that the recovery is highest when centrifugal concentrators are installed within the circulating load of the finer ball mill with a P80 of 60 µm. The modelling indicated that for a proposed 35 t/d, an Knelson model XD20 could treat 28% of the secondary cyclone underflow and recover 1,500 kg/d containing 10% of the Pd, 36% of the Pt, and 14% of the Au.

A simulation was conducted based on this finding and application of the LCT results from the previous study to the remaining flotation feed. The results of the simulation, presented in Figure 13-11 - when compared to the actual LCT result achieved - did not result in an improvement in overall Pd, Pt, or Au recovery for this sample, containing 8 g/t combined Pd+Pt+Au.

Figure 13-11: Simulation of Gravity Separation Results Compared to LCT-2 Results

Stream	Mass flow		Grade					Rougher Flotation Recovery (%)				
	(dmt/h)	(%)	Cu (%)	Ni (%)	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu	Ni	Pd	Pt	Au
Mill Feed	35	100	0.76	0.27	3.94	3.65	1.64	100.0	100.0	100.0	100.0	100.0
Gravity Concentrate	0.06	0.18	2.98	4.61	221	736	128	0.7	3	10	36	14
Flotation Feed	34.94	99.82	0.76	0.27	3.55	2.34	1.41	99.3	97.0	90.0	64.0	86.0
Rougher Concentrate	5.60	16.00	4.66	1.38	19.14	12.06	6.84	98.3	80.5	77.7	52.9	67.2
Cleaner Concentrate	1.93	5.50	13.34	3.82	51.31	31.81	16.88	96.6	76.4	71.6	47.9	56.7
Grav/Clnr Conc. Combined	1.99	5.68	13.01	3.84	56.64	53.95	20.38	97.3	79.4	81.6	83.9	70.6
LCT-2 Concentrate (Baseline)		5.5	13.56	3.76	57.8	56.5	19.69	97.3	75.1	82.4	85.4	72.7

(Source: Blue Coast PJ5313 Report P.22; Table 14)

Elemental Scan and Deleterious Elements

Elemental scans were initially performed on the LCT-2 Concentrate, as shown in Figure 13-12.

Figure 13-12: Elemental Scan of LCT-2 Concentrate

Analyte Symbol	Units	Analysis	LCT Assay	Analyte Symbol	Units	Analysis	LCT Assay
Ag	ppm	>100		Na	%	0.24	
Au	g/t		19.7	Nb	ppm	2.8	
Al	%	1.29		Nd	ppm	6.2	
Al2O3	%	2.55		Ni	%	>5000	0.0376
As	ppm	2670		Pb	ppm	1490	
B	ppm	<1		Pd	g/t		57.8
Ba	ppm	33		Pr	ppm	1.5	
Be	ppm	0.4		Pt	g/t		56.5
Bi	ppm	222		Rb	ppm	9	
Ca	%	0.94		Re	ppm	0.103	
Ce	ppm	13.9		S	%		0.294
Co	ppm	>500		Sb	ppm	4.2	
Cr	ppm	100		Sc	ppm	8	
Cs	ppm	0.4		SiO2	%	7.84	
Cu	ppm	>10000	13.6%	Se	ppm	177	
Dy	ppm	1.1		Sm	ppm	1.1	
Er	ppm	0.7		Sn	ppm	11	
Eu	ppm	0.37		Sr	ppm	35	
Fe	%	31.6		Ta	ppm	0.2	
Ga	ppm	3.7		Tb	ppm	0.2	
Gd	ppm	1.3		Te	ppm	130	
Hf	ppm	0.6		Th	ppm	3.1	
Hg	ppb	<10		TiO2	%	0.226	
Ho	ppm	0.3		Tl	ppm	2.76	
In	ppm	3.9		Tm	ppm	0.1	
K	%	0.13		U	ppm	1	
La	ppm	7.4		V	ppm	43	
Li	ppm	5.4		W	ppm	1	
Lu	ppm	0.1		Y	ppm	<1	
Mg	%	0.61		Yb	ppm	0.8	
MgO	%	1.09%		Zn	ppm	8580	
Mn	ppm	332		Zr	ppm	38	
Mo	ppm	34		LOI	%	12.15	

Source: Blue Coast PJ5219 Report P.30; Table 16

There were no elements identified that would impact the ability of the concentrate to be processed at a smelter. The only element that may be penalized is arsenic (As), at 2,670 ppm, which is close to the 2,500 ppm threshold.

A subsequent program evaluated the potential of gravity recoverable concentrate. Samples of open circuit flotation cleaner concentrate and gravity concentrates were also analyzed. The results are presented in Figure 13-13.

Figure 13-13: Elemental Scan of Flotation and Gravity Concentrates (Source: Blue Coast PJ5313 Report P.19/20; Table 12/13)

Analyte Symbol	Units	F-6 Flot Conc	F-7 Flot Conc	F-8 Flot Conc	F-9 Flot Conc	F-8 Grav Tip	F-9 Grav Tip
Ag	ppm	82.41	82.24	81.44	75.88	440.99	242.38
Au	g/t	13.9	13	12.2	11.4	944.7	320.6
Al	%	1.31	1.35	1.28	1.72	2.83	2.24
As	ppm	2741.61	2021.39	2701.55	2489.05	24001.29	21041.79
Ba	ppm	92.88	104.04	98.75	114.22	101.01	79.14
Be	ppm	<0.2	<0.2	<0.2	<0.2	9.81	9.29
Bi	ppm	157.54	163.59	141.81	145.26	979.38	702.9
Ca	%	0.84	0.95	0.72	1.17	1.73	1.14
Co	ppm	1415.46	1218.12	1357.7	1411.16	4199.69	3651.32
Cr	ppm	636.41	626.02	618.85	564.65	137	103.18
Cu	%	11.9	11.57	10.67	9.73	3.05	3.09
Fe	%	38.68	40.07	41.22	36.53	30.99	33.96
Ga	ppm	31.49	33.58	31.29	32.2	<100	<100
Hf	ppm	<20	<20	<20	<20	<100	<100
Hg	ppm	<20	<20	<20	<20		
In	ppm	<3	<3	<3	<3	<100	<100
K	%	0.14	0.15	0.14	0.18	0.2	0.16
Li	ppm	7.03	6.97	6.55	9.21	<10	<10
Mg	%	0.49	0.47	0.43	0.61	1.34	1.04
Mn	ppm	361.19	353.35	347.09	412.41	2136.14	1898.23
Mo	ppm	52.67	51.76	52.79	46.69	<5	<5
Na	%	0.51	0.63	0.57	0.68	<.05	<.05
Nb	ppm	<10	<10	<10	<10	<50	<50
Ni	%	3.31	3.18	3.11	2.75	4.98	4.56
P	%	0.09	0.09	0.08	0.1	0.07	0.06
Pb	ppm	1009.54	997.9	982.22	973.13	5243.23	3867.21
Pd	g/t	42.9	35.7	37.7	37.3	354.2	288.6
Pt	g/t	20.2	13.5	12.7	13	944.7	812.3
Rb	ppm	<20	<20	<20	<20	<100	<100
Re	ppm	<20	<20	<20	<20	<100	<100
S	%	27.59	27.59	27.85	25.67	15.81	16.26
Sb	ppm	29.41	22.07	24.91	24.38	84	62.49
Se	ppm	163.49	154.67	155.21	150.94	119.3	143.87
Sn	ppm	10.27	<10	<10	10.66	<50	<50
Sr	ppm	39.27	43.32	39.93	50.23	39.09	22.5
Ta	ppm	21.63	18.8	21.58	19.11	<50	<50
Te	ppm	155.39	148.06	132.89	127.53	938.85	719.98
Ti	%	0.12	0.11	0.11	0.14	2.83	2.61
Tl	ppm	4.86	2.96	5.27	4.75	<10	<10
V	ppm	41.6	38.94	38.16	50.32	138.49	110.65
W	ppm	<10	<10	<10	<10	87	<50
Zn	ppm	7905.03	7887.02	7644.36	7315.21	2017.48	1962.1
Zr	ppm	57.57	64.08	60.91	68.11	159.08	129.49

The flotation concentrate elemental analysis compared well with the LCT-2 results, with no elements at levels that would impact their ability to be processed at a smelter. As with the LCT, As averaged around the threshold value of 2,500 ppm, and may incur a minor penalty.

The samples from the gravity concentrate tip were elevated in Pt, Pd, and Au, and therefore had elevated levels of As at over 2%, and levels of Bi and Te of nearly 1,000 ppm, which are the components of the precious metal minerals. This concentrate mass is very small; therefore, this material would either be a specialty product for a smelter, or it would be re-combined with the flotation concentrate for treatment at a smelter. Once recombined with the concentrate, the elemental concentrations would be similar to the LCT-2 analysis.

13.5 MINERAL SENSING AND SORTING

Preconcentrating using ore sorting to reduce the amount of material that would have to be trucked and processed was evaluated by SRK in 2020.

For ore sorting to be feasible, the ore must be heterogeneous to differentiate valuable versus reject material. SRK used an NSR methodology to evaluate the heterogeneity of a deposit based on drill core data. Using 2020 NSR data, each drill core interval was compared against a 20-ft composite (based on the expected bench height).

The distribution of NSR of the sample interval versus the composite was then compared to assess its heterogeneity. In material contained in the resource above cut-off grade, 45.5% of the sample intervals were identified of having an NSR value of less than $45/tonne. The value of $45/tonne was selected as minimum value required as it is the estimated cost to truck and process one tonne of material. For material outside the resource, because it is below the cut-off grade, 30.6% of the intervals were identified as having an NSR value of greater than the $45/tonne required to process. This distribution indicated the potential for sorting either waste or resource material and justified advancing to testing.

To evaluate whether the ore sorting technologies available could separate the high and low NSR value material, specimens which displayed heterogeneity in NSR were selected from drill core and sent to Steinert GmbH facilities in Kentucky. 160 specimens were selected across six samples (20 from within the resource cut-off grade, and 30 from outside the resource cut-off grade.) Samples were tested using induction, x-ray transmission (XRT), and lasers (the last for particle sizing to assist the other techniques). Steinert established that the best results were a proprietary simulation that used all three of the technologies.

The samples were then shipped to ALS Global's testing laboratory in Kamloops, BC, for assay of precious metals and multi-element induced couple plasma mass spectroscopy (ICP). Results are shown in the table presented in Figure 13-14.

Figure 13-14: Test Results of Specimens Evaluated at Steinert GmbH (Source: XPS, 2023)

Denison Specimens within Resource as defined by Cut-Off Grade															
Stream	Count		Assay						Distribution %						
	Number	%	Cu %	Ni %	S %	Pd g/t	Pt g/t	Au g/t	Cu	Ni	S	Pd	Pt	Au
Product	28	70	1.86	0.89	4.12	7.69	5.04	2.88	98.2	97.1	98.6	98.7	96.5	97.5
Waste	12	30	0.08	0.06	0.14	0.24	0.42	0.17	1.8	2.9	1.4	1.3	3.2	2.5
Feed	40	100	1.33	0.64	2.92	5.45	3.66	2.06	100	100	100	100	99.7	100
Denison Specimens outside Resource as defined by Cut-Off Grade															
Stream	Count		Assay						Distribution %						
	Number	%	Cu %	Ni %	S %	Pd g/t	Pt g/t	Au g/t	Cu	Ni	S	Pd	Pt	Au
Product	41	34.2	0.33	0.08	0.51	0.56	1.14	0.39	74.2	63.6	73.7	81.7	87.1	81.1
Waste	79	65.8	0.06	0.02	0.1	0.06	0.09	0.05	25.8	36.4	26.3	18.3	12.9	18.9
Feed	120	100	0.15	0.05	0.24	0.23	0.45	0.16	100	100	100	100	100	100

Approximately 96% of the Au, Pt, Pd, Cu, and Ni metals are contained in 70% of the mass, indicating that 30% of the mass may be rejected with limited loss of metal. In sub-cut-off material, 34% of the mass contains 74% of the Cu, 64% of the Ni, and over 80% of the PGM. If pre-concentration can be applied to the sub-cut-off grade material, the opportunity exists to extract additional value.

To determine obtainable separations and evaluate the potential economic benefit of preconcentration, piloting of a bulk sample evaluation, along with estimates of capital and operating costs, are required.

13.6 EVALUATION OF TAILINGS REACTIVITY

The potential of acid drainage from process tailings were initially evaluated by Blue Coast using acid base accounting (ABA) and net acid generation (NAG) tests. A composite representing the process tailings was produced by bending the rougher and cleaner one tails from LCT-2 in the appropriate ratio. The results of these tests are shown in the tables presented in Figure 13-15 and Figure 13-16.

Figure 13-15: ABA Test Results (Source: Blue Coast PJ5219 Report P.36; Table 23)

Sample Source	Neutralizing Potential (NP) (kg $CaCO_3$/t)	MPA (kg $CaCO_3$/t)	NNP (kg $CaCO_3$/t)	Paste pH
LCT-2 Combined Tails	49	17.8	31	7.9

Figure 13-16: AG Test Results (Source: Blue Coast PJ5219 Report P.36; Table 24)

Sample Source	Net Acid Generation @ pH 4.5 (kg H_2SO_4/t)	Net Acid Generation @ pH 7.0 (kg H_2SO_4/t))	pH
LCT-2 Combined Tails	<0.01	<0.01	10.2

A shake flask extraction test was also performed to evaluate the potential of mobilization of metals contained in the tails at neutral pH. A separate tails sample was mixed at a distilled water to tails ratio of 3:1 and was allowed to react for 24 hours. The filtrate extracted from this test was analyzed using ICP. The analysis of the leachate is presented in Figure 13-17.

Figure 13-17: Shake Flask Metal Mobilization Test Results (Source: Blue Coast PJ5219 Report P.36; Table 25)

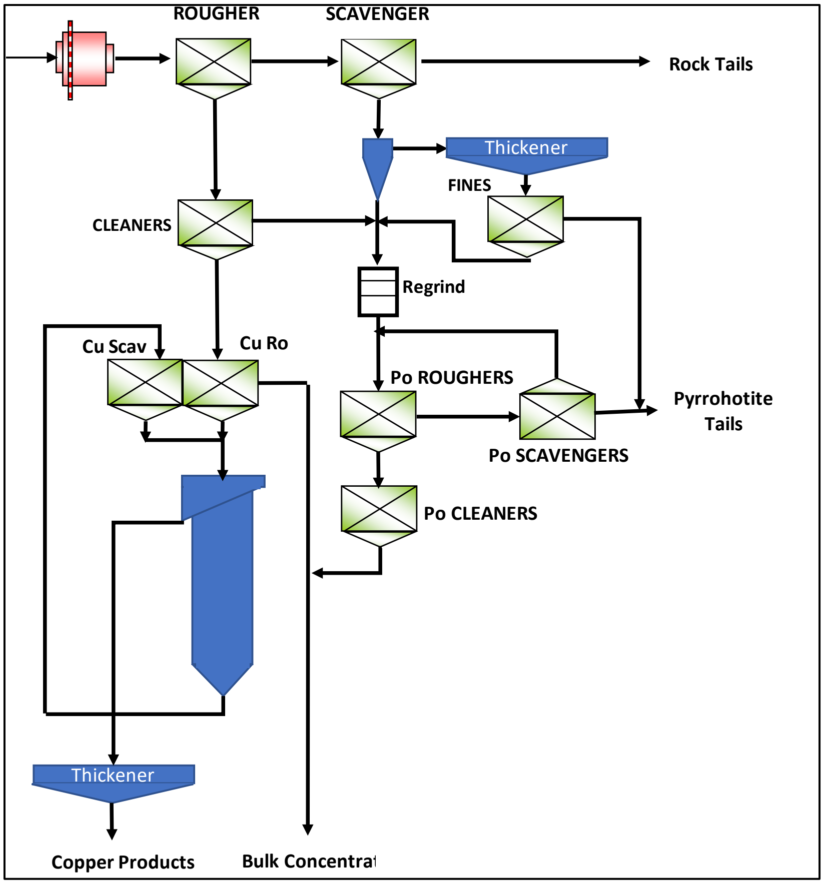
Element Symbol	Units	Concentration
Ag	mg/L	<0.000050
Al	mg/L	0.132
Ar	mg/L	0.0363
B	mg/L	0.022
Ba	mg/L	0.0067
Be	mg/L	<0.00050
Bi	mg/L	<0.00050
Ca	mg/L	21.0
Cd	mg/L	<0.000050
Co	mg/L	0.0017
Cr	mg/L	<0.00050
Cu	mg/L	<0.0010
Fe	mg/L	<0.030
Hg	mg/L	<0.000050
K	mg/L	7.81
Li	mg/L	<0.0050
Mg	mg/L	2.99
Mn	mg/L	0.00278
Mo	mg/L	0.0282
Na	mg/L	4.65
Ni	mg/L	0.00664
P	mg/L	<0.30
Pb	mg/L	<0.00010
Sb	mg/L	0.00108
Se	mg/L	0.00160
Si	mg/L	1.63
Sn	mg/L	<0.00050
Sr	mg/L	0.0591
Ti	mg/L	<0.010
Tl	mg/L	<0.00010
U	mg/L	0.000027
V	mg/L	<0.0010
Zn	mg/L	<0.010

The ABA test and the NAG test indicate that the tailings from the process are unlikely to generate acid, and the shake flask extraction tests indicate that that the contained metals have a low level of mobility at neutral pH.

13.7 VALE CLARABELLE MILL

The option being evaluated is selling the ore to an existing mining company for processing with their ores. Vale's Clarabelle Mill is an example of an existing processing plant: the process is shown in Figure 13-18.

Figure 13-18: Vale Clarabelle Process (Source: Vale 2021 Technical Report Figure 14-3; P158)



Some test work has been performed for processing 109 FW material using the current Clarabelle Mill process (refer to Section 13.4.1). The other resources, which represent 70% of the resource, are made up of extensions and remnants of historic mining operations and are primarily contact type ore. The response of this contact ore through the Clarabelle Mill will depend primarily on the grade of Ni in the material and the amount of pyrrhotite that must be rejected.

Testing will be conducted on representative samples to determine the response of the contact ores and any additional FW material. In the absence of test results, the current performance of the Clarabelle Mill can provide an indication of the potential metallurgical results.

The most recent results from Vale are available in 7-April-2022 – C, Gould et. al. - Vale- Technical Report Summary, Sudbury Property, Ontario Operations, Canada. Past performance of Cu and Ni recovery is presented in Figure 13-19, and projections of Pd, Pt, and Au recoveries are shown in Figure 13-20.

Figure 13-19: Vale Clarabelle Mill Reported Recovery (Source: Vale 2021 Technical Report Table 10-1; P124)

Year	Clarabelle Mill Actual Data				
	Feed			Concentrate	
	Cu (%)	Ni (%)	S (%)	Cu Rec (%)	Ni Rec (%)
2016	2.04	1.46	9.7	97.4	84.9
2017	2.24	1.60	10.0	97.1	83.8
2018	2.19	1.60	10.0	97.1	85.0
2019	2.26	1.46	9.2	97.0	85.2
2020	1.88	1.27	7.9	96.4	84.3
2021	1.90	1.26	8.1	95.7	84.0

Note: Source of data is Table 10-1 Vale 2022 Technical Report

Figure 13-20: Vale Mill Recovery Projections (Source: Vale 2021 Technical Report Table 10-3

Metal	Mill Recovery to Bulk Concentrates (%)
Cu	94.6
Ni	84.6
Co	79.2
Pt	79.3
Pd	83.8
Au	74.4

Note: Data sourced from Table 10-3 Vale 2022 Technical Report

13.8 METAL RECOVERY

Metal recovery estimates are shown in Figure 13-21 for the processing of the resource through the Clarabelle Mill. The estimated recoveries require confirmation with additional test work.

Figure 13-21: Projected Metallurgy from Resource (Source: XPS, 2023)

Total Concentrate Recoveries			**Reference**
Cu	<0.21	=79.6/0.2*CuH	Vale tests
	>0.21<0.76	=79.6+6.4/0.55*(CuH-0.21)	Blue Coast tests
	>0.76	96	Vale Technical Report
Ni	<.22	=51.6/.22*(NiH)	Vale tests
	>.22<.29	=51.6+24.5/0.07*(NiH-0.22)	Blue Coast tests
	>.29<1.5	=75.1+9.9/1.21*(NiH-0.29)	Vale Technical Report
	>1.5	85	Vale Technical Report
Co	Ni Rec - 3%		Vale Technical Report
PGM Recoveries from Ni Feeds (Ni>0.22)			
Pt	Ni Rec * 0.9		Average of tests
Pd	Ni Rec * 0.9		Average of tests
Au	Ni Rec * 0.8		Average of tests
PGM Recoveries from High PGM (Pt+Pd+Au>5 g/t) Low Ni Feeds (Ni<0.22)			
Pt	Cu Rec*0.5		
Pd	Cu Rec*0.5		
Au	Cu Rec*0.5		

13.9 RECOMMENDATIONS

Based on the results of the test work completed at the time of publication of this report, the following are recommended to complete further testing:

1. To complete the evaluation of selling ore to an existing mining company
    a. Evaluation of representative samples from untested zones.
    b. Evaluation of blending to enhance precious metal recovery.
2. To evaluate other opportunities to enhance value
    a. Evaluation of impact of gravity separation on coarse primary grind.
    b. Evaluation of ore sorting through testing of representative samples.
    c. Evaluation of processing through the potential Shakespeare Mill with necessary plant modifications to handle additional concentrate.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

Completion of the updated mineral resource estimates (MREs) for the Property involved the assessment of a drill hole database, which included all data for surface drilling completed through the end of 2017, as well as three-dimensional (3D) mineral resource models (resource domains), 3D models of all mined-out areas (open pit and underground), 3D models of cross-cutting dykes, a recent topographic surface, and available written reports.

The Inverse Distance Squared (ID2) calculation method restricted to mineralized domains was used to interpolate grades for Ni (%), Cu (%), Co (%), Pt (g/t), Pd (g/t), and Au (g/t) into block models.

Indicated and Inferred mineral resources are reported in the summary tables in Section 14.11. The current MRE takes into consideration that the Projects deposits may be mined by open pit and underground mining methods.

14.2 DRILL HOLE DATABASE

In order to complete the MRE for the Property, a database comprising a series of comma delimited spreadsheets containing surface and underground drill hole information was provided by Magna. The database included hole location information (local grid coordinates, in feet), survey data (final depth in feet), assay data (from and to in feet), lithology data, and specific gravity data. The data in the assay table included assays for Ni (%), Cu (%), Co (%), Pt (g/t), Pd (g/t), and Au (g/t), as well as Ag (g/t), Rh (ppm), S (%) and Fe (%). Note that not all assay samples had values for Pt, Pd, Au, Ag, or Rh. Ag and Rh were the least-analysed elements and are not included in the MRE (see Section 14.5 for a summary of assay data).

After review of the database, data were imported into GEOVIA GEMS version 6.8.3 software (GEMS) for statistical analysis, block modeling, and resource estimation.

The original database provided by Magna included data for 4,719 surface and underground drill holes. The database was reduced to include only data for surface and underground drill holes completed within the current property boundary. Thus, the database used for the current MRE comprises data for 3,836 surface and underground drill holes, totalling 1.57 million ft (478,000 m) (Figure 14-1 and Figure 14-2). The database includes 89,257 assay intervals, totalling 622,082 ft (189,611 m) (see Section 14.4).

The database was checked for typographical errors in drill hole locations, down-hole surveys, lithology, assay values, and supporting information on source of assay values. Overlaps and gapping in survey, lithology, and assay values in intervals were checked. Gaps in the assay sampling and unsampled elements were assigned a grade value of 0.0001 for Co, Pt, Pd, and Au.

Figure 14-1: Plan View – Distribution of Surface and Underground Drill Holes on the Crean Hill Property in Local Mine Grid

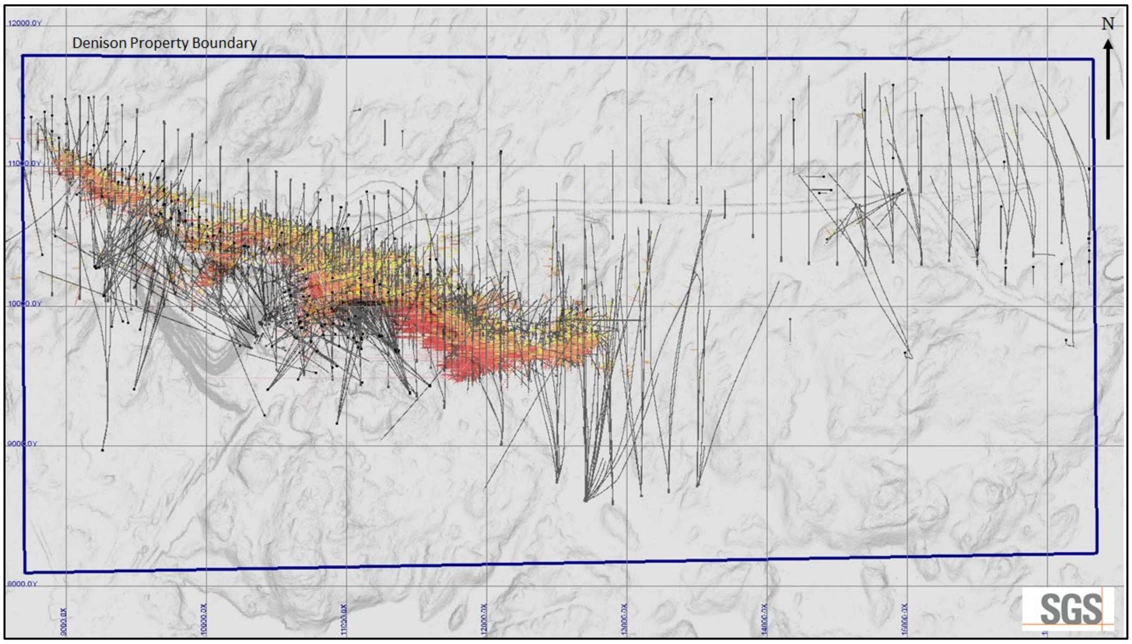
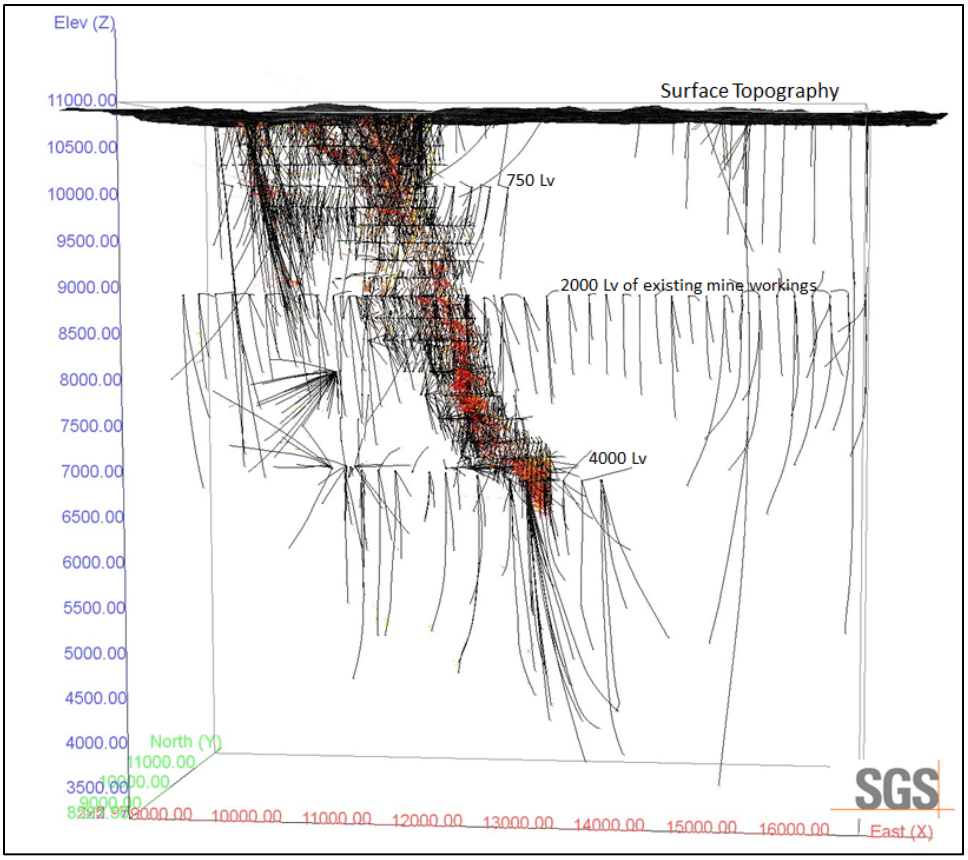


Figure 14-2: Isometric View Looking North – Distribution of Surface and Underground Drill Holes on the Crean Hill Property in Local Mine Grid



14.3 MINERAL RESOURCE MODELLING AND WIREFRAMING

The Author was provided with a total of 20 3D wireframe models (mineral domains) of mineralization, to be used for the current MRE (Figure 14-3). Solids were developed in Leapfrog using a minimum $50/ton metal value calculated using metal prices and recoveries outlined in Table 14-1 (WSP, 2020). No minimum thickness was applied, and all drillholes were referenced regardless of the date they were drilled. Adjustments were made to the solids to account for underground mapping. The domains of mineralization interpreted for each area were generally contiguous; however, due to the nature of the mineralization, there are portions of the wireframe that contain zones of poor mineralization yet are still within the mineralizing trend. Several of the domains have minor overlap with other domains. The overlap was to ensure there was no gap in the block model and to account for changing parameters within the deposit. The final merged model removes any overlap with the blocks.

The Author has reviewed the mineral domains on section. In the Author's opinion the models provided are very well constructed and fairly accurately represent the distribution of the high-grade mineralization within the Property. No re-modeling of the deposits is recommended at this time. Limited sporadic mineralization exists outside of these wireframes, as well as along strike and at depth. With additional drilling, some areas of scattered mineralization may get incorporated into the mineral domains.

The Crean Hill deposit generally strikes 85° to 110° and dips / plunges steeply south, with the exception of the 101 Zone which strikes at 40° and dips near vertical. The mineral domains extend for roughly 3,100 ft (945 m) along strike and reach a maximum depth of 5,000 ft (1,524 m) below surface.

The Author was also provided with six 3D dyke models (olivine diabase dykes and quartz diabase dykes [trap dykes]; Figure 14-4), 3D models of the surface and underground mined out areas (voids-out; Figure 14-5), and a 3D surface model of the current topography (Figure 14-6). The topographic digital terrain model was generated using LiDAR topographic data collected by Loncan.

Table 14-2 summarizes the mineral domains, dykes, and mined areas. All mineral domains are clipped to topography and property boundary.

Table 14-1: Input Values Used to Determine Resource Model Base Case Cut-off Grade

Metal	Value	Units
Metal Prices		
Cu	2.75	US$/lb
Ni	6.5	US$/lb
Co	15	US$/lb
Pt	1000	US$/oz
Pd	1450	US$/oz
Au	1500	US$/oz
Ag	16	US$/oz
Recoveries		
Cu	95.5	%
Ni	78	%
Co	0	%
Pt	69.2	%
Pd	68	%
Au	67.7	%
Ag	50	%

Figure 14-3: Crean Hill Mineral Domains – Isometric View Looking North (A) and Plan View (B)

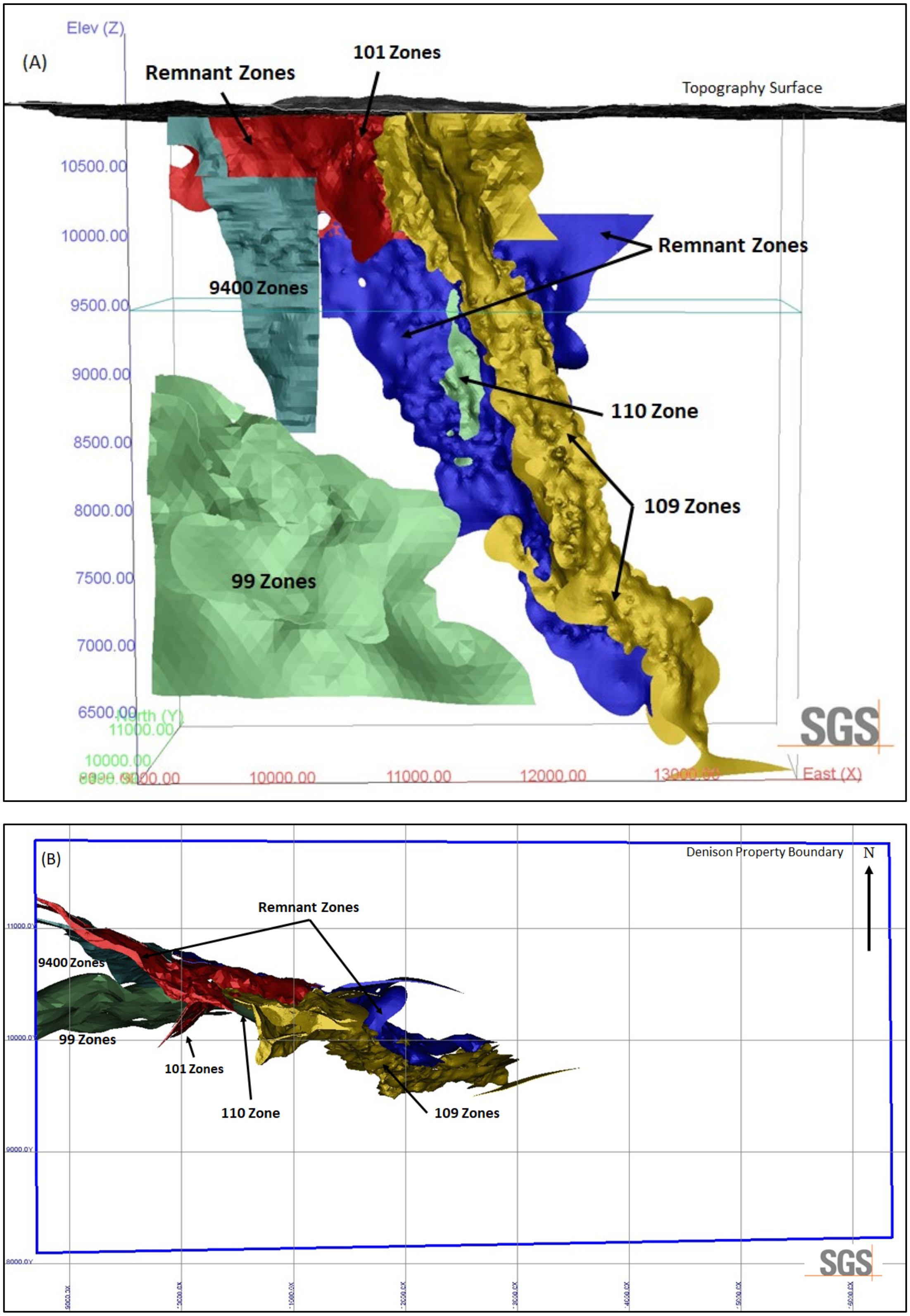


Figure 14-4: Crean Hill Diabase Models – Isometric View Looking North

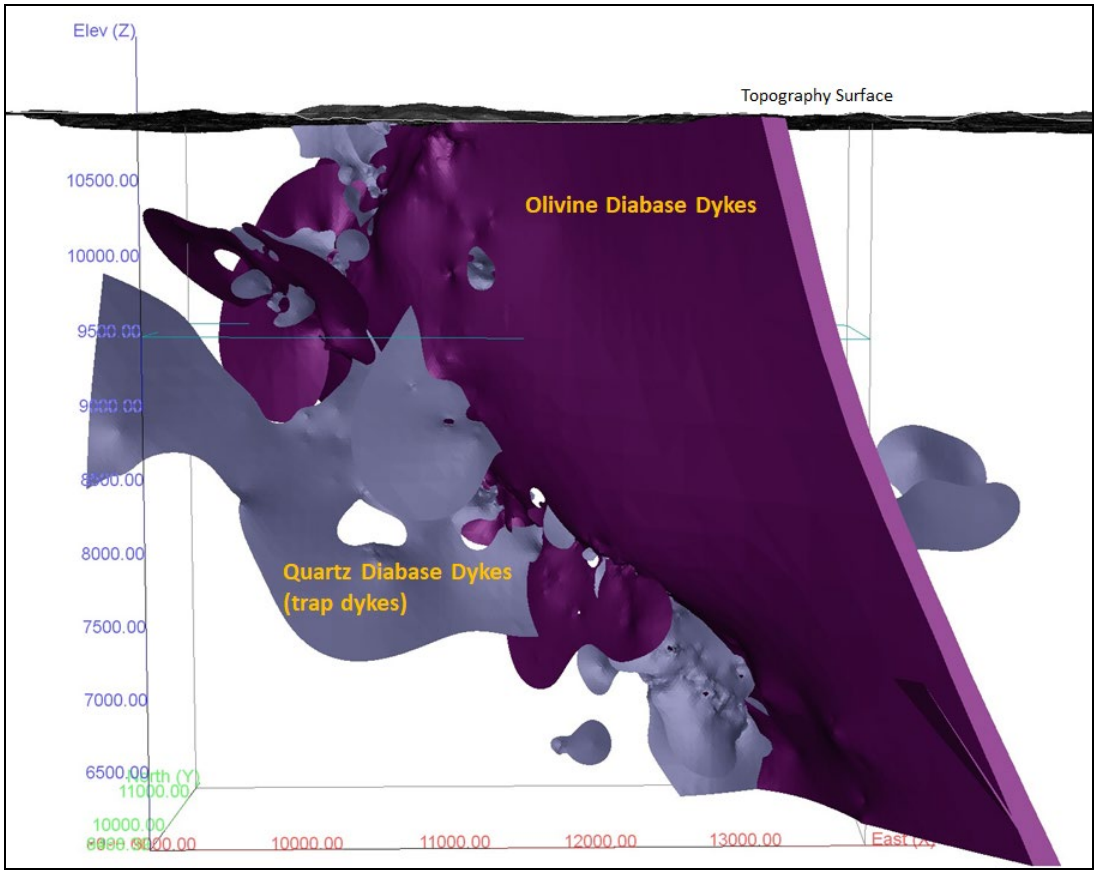


Figure 14-5: Mined Out Stopes and Pits – Isometric View Looking North

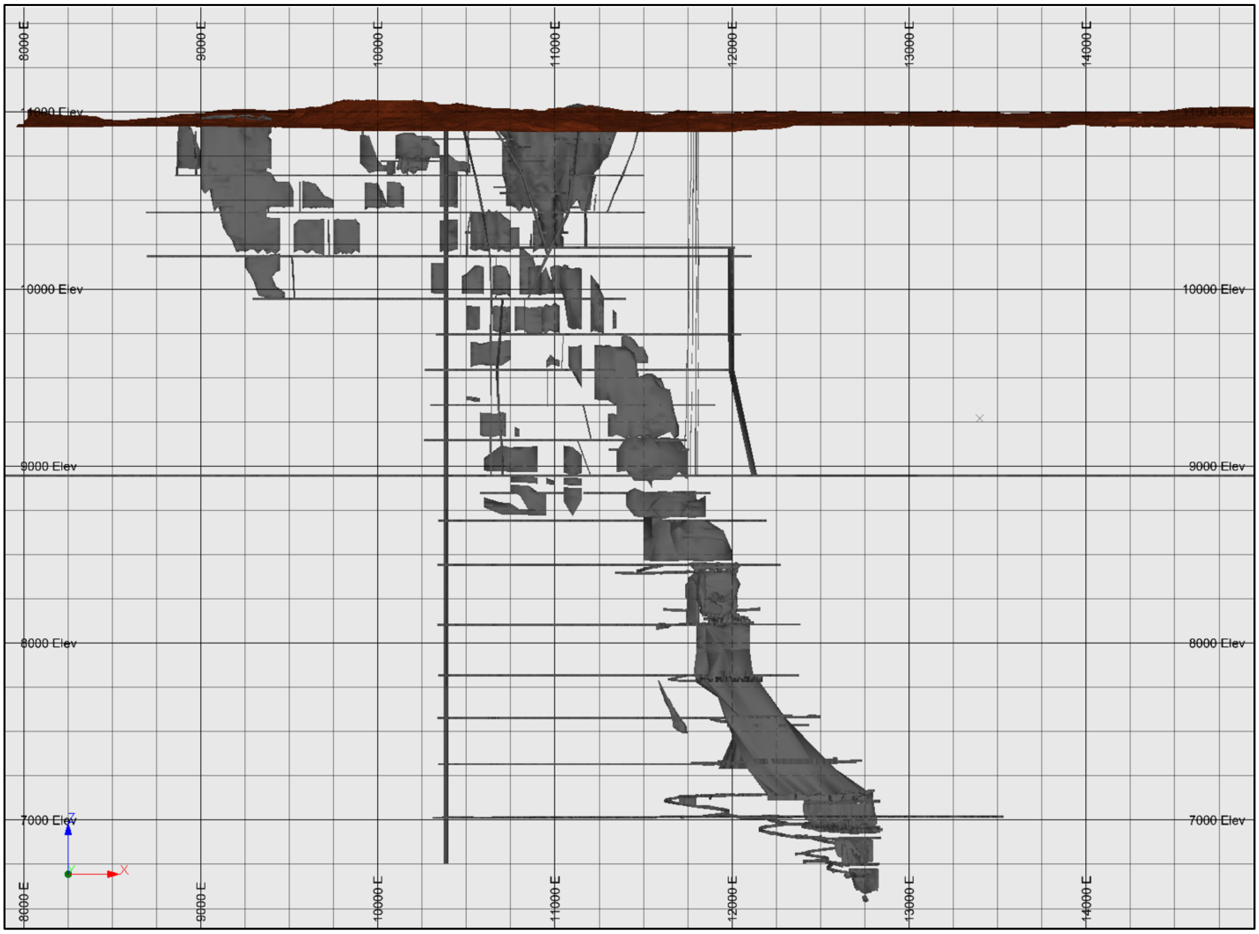


Figure 14-6: Crean Hill Property Digital Terrain Model – Plan View

Table 14-2: Crean Hill Property Domain Descriptions

Domain	Rock Code	Block Rock Code	Solid Precedence	Volume (ft^3)	Density (Ton/ft^3)	Ton	Tonnes
101 Zone	101	400	8	13,309,098	0.096	1,277,673	1,159,086
109 FW Below 10000	109FWB	1050	4	62,130,585	0.107	6,647,973	6,030,939
109 FW_2 Below 10000	109FW2B	1100	4	11,165,718	0.096	1,071,909	972,419
109 FW_4 Below 10000	109FW4B	1150	4	2,108,467	0.096	202,413	183,626
109 W Below 10000	109WB	1200	4	31,601,530	0.094	2,970,544	2,694,832
109fw_2017	109FW	850	4	40,637,928	0.092	3,738,689	3,391,682
109W_2 Below 10000	109W2B	1250	4	93,345,846	0.096	8,961,201	8,129,465
110 Below 10000	110BELOW	1300	7	3,471,500	0.097	336,736	305,481
115 Below 10000	115BELOW	1350	7	460,280	0.092	42,346	38,415
9400 FW_Ext	9400FW	600	5	2,804,116	0.095	266,391	241,666
9400_2017	94002017	950	5	46,278,375	0.094	4,350,167	3,946,405
99-1_50	991	100	3	140,291,855	0.092	12,906,851	11,708,898
99-2_50	992	200	3	51,009,380	0.092	4,692,863	4,257,294
99-FW_50	99FW	300	3	15,222,391	0.094	1,430,905	1,298,095
Rem Below 10000	REMBELOW	1400	6	181,004,532	0.094	17,014,426	15,435,228
Rem_10000z	REMNANT	500	6	80,268,283	0.094	7,545,219	6,844,907
Rem_109W	109W	700	6	41,947,607	0.094	3,943,075	3,577,098
Rem_109W	109W2	750	6	64,004,144	0.094	6,016,390	5,457,977
Rem_South_East	REMBELOW	1400	6	3,246,089	0.094	305,132	276,811
REM_South_West	REMNANTW	550	6	9,832,449	0.095	934,083	847,386
Total				**894,140,173**		**84,654,984**	**76,797,710**
Dykes							
OD 2	DYKES	3	2		0.094		
OD 3	DYKES	3	2		0.094		
OD 4	DYKES	3	2		0.094		
OD 1	DYKES	3	2		0.094		
Trap 2	DYKEST	4	2		0.087		
Trap 1	DYKEST	4	2		0.087		
Voids - Out	STOPES	2	1		0.0624		
Waste	WASTE	1	9		0.088		

14.4 BULK DENSITY

Most diamond drill core samples completed by Loncan from 2015-2017 were subject to specific gravity measurement, by measuring dry and submerged sample weight (WSP, 2020). Each sample was allowed to dry fully after being cut and weighed on top of the balance. The sample was then placed in a mesh basket suspended from a free-hanging hook below the balance and weighed submerged in water. The water was kept at approximately 20°C using a heater / agitator.

Specific gravity is calculated as follows:

- SG = Dry weight / (Dry weight – Wet weight) × 0.998

Where 0.998 is a temperature correction for at water at 20°C. A conversion factor of 0.0312 ton/ft^3 equals 1 tonne/m^3.

Density is expressed as short tons per cubic ft (ton/ft^3).

To ensure high-quality data, the balance was checked with reference weights prior to each day of density determinations, and a density determination was performed on a reference rock sample.

For samples without density measurements—which includes all historical data post-1968—density was determined using an Alcock regression formula. Where Cu, Ni, and S assays were available, density was calculated as follows:

- SG = 100 / (100 / 2.88 + 0.0166 × %Cu - 0.1077 × %Ni - 0.328 × %S)

For samples drilled before 1968 where only Cu and Ni assay results are available, SG was calculated as below:

- Density = 2.80 + 0.02 × %Cu + 0.20 × %Ni

These formulae were developed for semi-massive to massive contact Ni-Cu sulphide deposits. They are known to underestimate the density of most felsic rocks and overestimate the density of most mafic rock types outside the SIC. In these formulae, sulphides contribute significantly to the density.

Based on a review of the available density data, it was decided that a fixed value be used for each resource model, dyke model, stope / mined-out model, and waste (Table 14-1).

A density of 0.094 ton/ft^3 has been assigned to the Olivine Diabase dykes, while Trap dykes were assigned a value of 0.087 ton/ft^3 (WSP, 2020). All other waste has been assigned a density of 0.088 ton/ft^3.

The stopes have been assumed to be approximately 2/3 rock fill and 1/3 void space and were assigned a density of 0.0624 ton/ft^3 (WSP, 2020). Other mine workings (air raises, drifts, and escape ways) are assumed to be void pace and have been assigned a density of 0 ton/ft^3.

14.5 COMPOSITING

The assay sample database available for the current resource estimate included 89,257 assay intervals for a total of 622,082 ft (189,611 m) of drilling. Of this, 41,293 assays (281,409 ft or 85,773 m) occur within the deposit mineral domains. Of the 41,293 assays, all had Ni and Cu values; 24,864 had Pt values; 24,706 had Pd values; 20,340 had Au values; and 8,037 had Ag values. Silver is not estimated for this mineral resource. Unsampled precious metals were given a nominal value of 0.0001.

A statistical analysis was performed on the assay data from within the mineralized domains. The average length of assay sample intervals is 6.81 ft (2.08 m). Of the total assay population, approximately 80% are 10 ft (3.05 m) long or less. Approximately 91% of the samples are less than 14.8 ft (4.50 m) in length.

Sample intervals were composited into 10 ft (3.05 m) downhole intervals honouring the interpreted mineralization solids. The composites were extracted to point files for statistical analysis and capping studies. The constrained composites were grouped based on the mineral domain (rock code) of the constraining wireframe model.

Composites were generated starting from the collar of each hole, then constrained to the individual mineral domains. The constrained composites were extracted to point files for statistical analysis and capping studies.

A total of 28,451 composite sample points occur within the resource wire frame models. A statistical analysis of the composite data from within the mineralized domains is presented in Table 14-3. These values were used to interpolate grade into resource blocks.

Table 14-3: Statistical Analysis of the 10 ft (3.05 m) Composite Data from within the Crean Hill Mineral Domains

Variable	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
	All Domains					
Total # Assay Samples	28,451					
Average Sample Length	10 ft (3.05 m)					
Minimum Grade	0.00	0.00	0.00	0.00	0.00	0.00
Maximum Grade	11.90	16.98	0.55	177.30	102.22	42.63
Mean	0.72	0.66	0.03	0.39	0.37	0.14
Standard Deviation	0.94	0.84	0.03	2.07	1.57	0.65
Coefficient of variation	1.30	1.28	1.17	5.23	4.22	4.74
97.5 Percentile	3.59	2.91	0.12	2.26	2.52	0.89

14.6 GRADE CAPPING

A statistical analysis of the cumulative composite database within the deposit wireframe models (the "resource" population) was conducted to investigate the presence of high-grade outliers, which can have a disproportionately large influence on the average grade of a mineral deposit. High-grade outliers in the composite data were investigated using statistical data, histogram plots, and cumulative probability plots of the composite data. The statistical analysis was completed using GEMS.

After a review of the composites globally and by domain, it is the Author's opinion that minimal capping of high-grade composites to limit their influence during the grade estimation is necessary. Appropriate capping levels were chosen by metal and however it was decided, based on statistical analysis the same capping levels be applied to all domains. A summary of grade capping values within the mineralized domains is presented in Table 14-4. Capped composites are used for grade interpolation into the Crean Hill deposit block model.

Table 14-4: Composite Capping Summary of the Crean Hill Deposit Mineral Domains

Domain	Total # of Composites	Attribute	Capping Value	# Capped	Mean of Raw Composites	Mean of Capped Composites	CoV of Raw Composites	CoV of Capped Composites
All Domains	28,451	Ni %	7.5	2	0.72	0.72	1.30	1.30
		Cu %	9.5	14	0.66	0.66	1.28	1.26
		Co %	0.2	14	0.03	0.03	1.17	1.15
		Pt g/t	35.0	14	0.39	0.38	5.23	3.49
		Pd g/t	35.0	7	0.37	0.36	4.22	3.43
		Au g/t	19.0	4	0.14	0.14	4.74	4.35

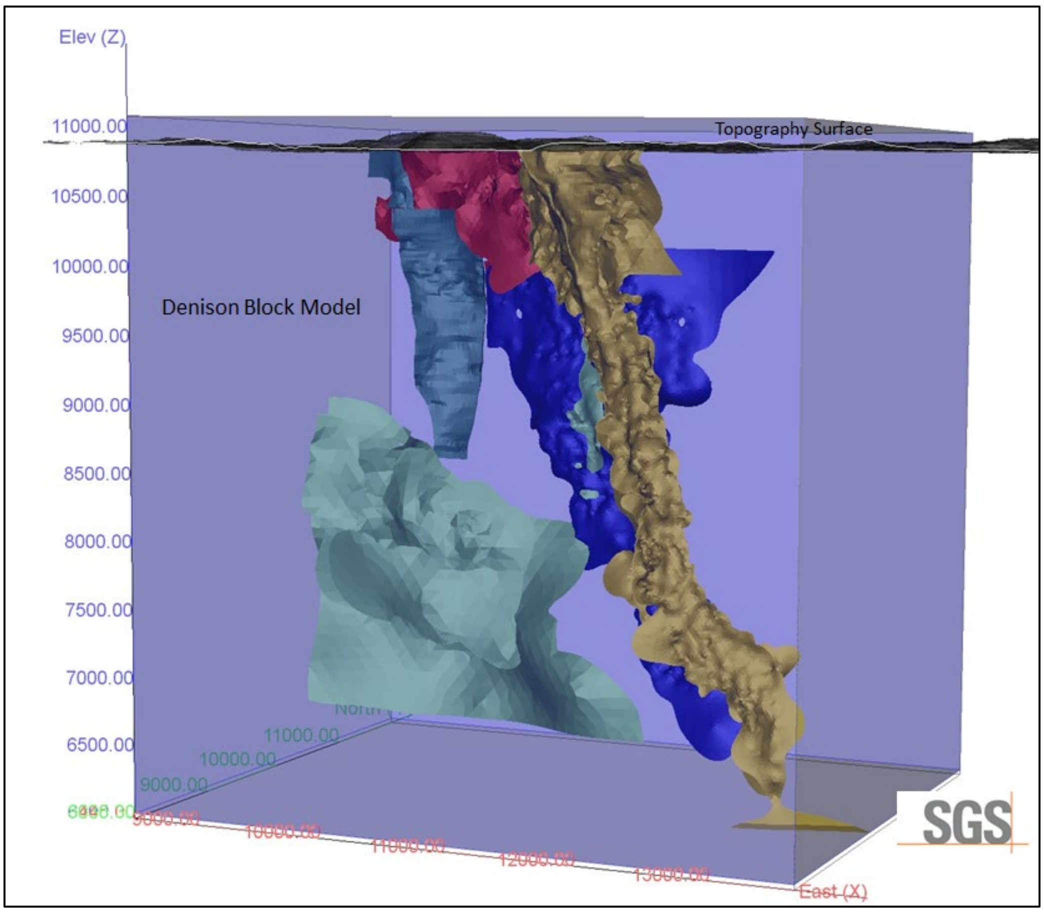
14.7 BLOCK MODEL PARAMETERS

The Property mineral domains are used to constrain composite values chosen for interpolation, and the mineral blocks reported in the estimate of the Mineral Resource. A block model within local mine grid coordinate space (no rotation; Table 14-5 and Figure 14-5) with block dimensions of 16.4 ft × 16.4 ft × 32.8 ft (5 m × 5 m × 10 m) in the x (east), y (north) and z (level) directions was placed over the grade shells. Only that portion of each block inside the shell was recorded (as a percentage of the block) as part of the MRE (% Block Model). The block size was selected based on borehole spacing, composite length, the geometry of the mineralized domains, and the selected starting mining method (Open Pit). At the scale of the Crean Hill deposit this provides a reasonable block size for discerning grade distribution, while still being large enough not to mislead when looking at higher cut-off grade distribution within the model. Surface topography was used to exclude blocks, or portions of blocks, that extended above the modeled bedrock surface.

Table 14-5: Deposit Block Model Geometry

Block Model	Crean Hill Property		
	X (North)	Y (East)	Z (Level)
Origin (Local Grid)	8445	8040	11120
Extent (blocks)	315	235	145
Block Size	16.4 ft (5 m)	16.4 ft (5 m)	32.8 ft (10 m)
Rotation (counterclockwise)	0°		

Figure 14-7: Isometric View Looking Southeast Showing the Crean Hill Deposit Mineral Resource Block Model and Mineralization Domains



14.8 GRADE INTERPOLATION

Nickel, copper, cobalt, platinum, palladium and gold were estimated for each domain in the Crean Hill deposit. Blocks within each mineralized domain were interpolated using composites assigned to that domain. To generate grade within the blocks, the inverse distance squared (ID2) interpolation method was used for all domains.

For all domains, the search ellipse used to interpolate grade into the resource blocks was interpreted based on orientation and size the mineralized domains. The search ellipse axes are generally oriented to reflect the observed preferential long axis (geological trend) of the vein structures and the observed trend of the mineralization down dip / down plunge (Table 14-6).

Three passes were used to interpolate grade into all of the blocks in the grade shells (Table 14-6). For Pass 1 the search ellipse size for all mineralized domains was set to 99 ft × 73 ft × 26 ft in the x-, y-, and z-direction, respectively. For Pass 2 the search ellipse size for each domain was set to 198 ft × 146 ft × 54 ft. For Pass 3 the search ellipse size was set to 329 ft × 230 ft × 78 ft. Blocks were classified as Indicated if they were populated with grade during Pass 1 and Pass 2 of the interpolation procedure. The Pass 3 search ellipse size was set to assure all remaining blocks within the wireframe (within the extent of the search ellipse) were assigned a grade. These blocks were classified as Inferred.

Grades were interpolated into blocks using 7-10 composites to generate block grades during Pass 1 (maximum of three sample composites per drill hole), 5-10 for Pass 2 (maximum of three sample composites per drill hole), and 3 -10 composites to generate block grades during Pass 3 (Table 14-6).

Table 14-6: Grade Interpolation Parameters by Domain

Parameter	99 Zones			101 Zones			All Other Zones		
	Pass 1	Pass 2	Pass 3	Pass 1	Pass 2	Pass 3	Pass 1	Pass 2	Pass 3
	Indicated	Indicated	Inferred	Indicated	Indicated	Inferred	Indicated	Indicated	Inferred
Calculation Method	Inverse Distance squared			Inverse Distance squared			Inverse Distance squared		
Search Type	Ellipsoid			Ellipsoid			Ellipsoid		
Principle Azimuth	100°			65°			115°		
Principle Dip	-55°			-65°			-55°		
Intermediate Azimuth	75°			35°			85°		
Anisotropy X – ft (m)	99 (30.2)	198 (60.4)	329 (100.3)	99 (30.2)	198 (60.4)	329 (100.3)	99 (30.2)	198 (60.4)	329 (100.3)
Anisotropy Y– ft (m)	73 (22.3)	146 (44.5)	230 (70.1)	73 (22.3)	146 (44.5)	230 (70.1)	73 (22.3)	146 (44.5)	230 (70.1)
Anisotropy Z– ft (m)	26 (7.9)	54 (16.5)	78 (23.8)	26 (7.9)	54 (16.5)	78 (23.8)	26 (7.9)	54 (16.5)	78 (23.8)
Min. Samples	7	5	3	7	5	3	7	5	3
Max. Samples	10	10	10	10	10	10	10	10	10
Min. Drill Holes	3	2	1	3	2	1	3	2	1

14.9 MINERAL RESOURCE CLASSIFICATION PARAMETERS

The Mineral Resource Estimate presented in this Technical Report was prepared and disclosed in compliance with all current disclosure requirements for mineral resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects. The classification of the current Mineral Resource Estimate into Indicated and Inferred is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves, including the critical requirement that all mineral resources “have reasonable prospects for eventual economic extraction”.

The current Mineral Resource is sub-divided, in order of increasing geological confidence, into Inferred and Indicated categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource. There are no Measured Mineral Resources reported.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

Interpretation of the word ‘eventual’ in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage ‘eventual economic extraction’ as covering time periods in excess of 50 years. However, for many gold or base metal deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

14.9.1 Indicated Mineral Resource

An ‘Indicated Mineral Resource’ is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource Estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

14.9.2 Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

14.10 MINERAL RESOURCE STATEMENT

The general requirement that all Mineral Resources have “reasonable prospects for eventual economic extraction” implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade, taking into account extraction scenarios and processing recoveries. To meet this requirement, the Author considers that the Denison deposit mineralization is amenable for open pit and underground extraction.

To determine the quantities of material offering “reasonable prospects for economic extraction” by an open pit, Whittle™ pit optimization software (version 4.7.1) and reasonable mining assumptions were used to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be “reasonably expected” to be mined from an open pit. Pit optimization was completed by SGS. The pit optimization parameters used are summarized in Table 14-6. A Whittle pit shell at a revenue factor of 1.0 was selected as the ultimate pit shell for the purposes of this MRE. The corresponding strip ratio is 10.6:1 and reaches a maximum depth below surface of approximately 1,320 ft (402 m) in the east and 1,250 ft (381 m) in the west. The optimized pit is limited to the property boundary.

The reader is cautioned that the results from the pit optimization are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the Property. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade. A selected base-case cut-off grade of 0.3% NiEq was used to determine the in-pit MRE for the Denison deposit.

To determine the quantities of material offering "reasonable prospects for economic extraction" by underground mining methods, reasonable mining assumptions to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from underground are used. Based on the size, shape, and orientation of the Deposit, it is envisioned that the Deposit may be mined using the longhole open stoping mining method (a bulk mining method that has long been utilized in the Sudbury region). The underground parameters used, based on this mining method, are summarized in Table 146. Based on these parameters, a selected base-case cut-off grade of 1.1% NiEq was used to determine the below-pit MRE for the Crean Hill deposit. The below-pit MRE is limited to a depth of ~4,500 ft (1,371.6 m) below surface.

The reader is cautioned that the reporting of the underground resources are presented undiluted and in situ (no minimum thickness), constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction. There are no underground mineral reserves reported at this time.

The current MRE for the Crean Hill deposit is presented in Table 14-7 and includes an in-pit and an underground (below-pit) Mineral Resource (estimated from the bottom of the 2022 pit) (Figure 14-8 and Figure 14-9).

Highlights of the Crean Hill deposit Mineral Resource Estimate are as follows:

- The in-pit Mineral Resource includes, at a base-case cut-off grade of 0.3% NiEq, 16,760,000 tonnes grading 0.53% Ni, 0.49% Cu, 0.02% Co, 0.48 g/t Pt, 0.37 g/t Pd and 0.25 g/t Au in the Indicated category, and 434,000 tonnes grading 0.43% Ni, 0.49% Cu, 0.02% Co, 0.29 g/t Pt, 0.14 g/t Pd and 0.07 g/t Au in the Inferred category.
- The below-pit Mineral Resource includes, at a base-case cut-off grade of 1.1% NiEq, 14,532,000 tonnes grading 0.96% Ni, 0.84% Cu, 0.03% Co, 0.88 g/t Pt, 1.02 g/t Pd and 0.54 g/t Au in the Indicated category, and 1,169,000 tonnes grading 0.61% Ni, 0.46% Cu, 0.02% Co, 0.64 g/t Pt, 1.09 g/t Pd and 0.21 g/t Au in the Inferred category.

Table 14-7: Whittle™ Pit Optimization Parameters

Parameter	Value	Unit
Nickel Price	$8.50	US$ per pound
Copper Price	$3.75	US$ per pound
Cobalt Price	$22.00	US$ per pound
Platinum Price	$1,000.00	US$ per ounce
Palladium Price	$2,000.00	US$ per ounce

Parameter	Value	Unit
Gold Price	$1,750.00	US$ per ounce
In-Pit Mining Cost	$2.50	US$ per tonne mined
Underground Mining Cost	$80.00	US$ per tonne mined
Transportation	$5.00	US$ per tonne milled
Processing Cost (incl. crushing)	$15.50	US$ per tonne milled
Treatment and Refining	$15.00	US$ per tonne milled
In-Pit General and Administrative (G&A)	$2.50	US$ tonne of feed
Underground General and Administrative	$7.00	US$ tonne of feed
Overall Pit Slope	55	Degrees
Nickel Recovery	78.0	Percent (%)
Copper Recovery	95.5	Percent (%)
Cobalt Recovery	56.0	Percent (%)
Platinum Recovery	69.2	Percent (%)
Palladium Recovery	68.0	Percent (%)
Gold Recovery	67.7	Percent (%)
Mining loss / Dilution (open pit)	5/5	Percent (%) / Percent (%)
Mining loss/Dilution (underground)	10/10	Percent (%) / Percent (%)

Table 14-8: Crean Hill Deposit In-Pit and Underground (Below-Pit) Mineral Resource Estimate, August 19, 2022

Cut-off Grade	Tonnes	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	NiEq %
In-Pit Mineral Resource Estimate								
Indicated								
0.3% NiEq	16,760,000	0.53	0.49	0.02	0.48	0.37	0.25	1.08
Inferred								
0.3% NiEq	434,000	0.43	0.49	0.02	0.29	0.14	0.07	0.82
Underground (Below-Pit) Mineral Resource Estimate								
Indicated								
1.1% NiEq	14,531,000	0.96	0.84	0.03	0.88	1.02	0.54	2.07
Inferred								
1.1% NiEq	1,170,000	0.61	0.46	0.02	0.64	1.09	0.21	1.41

(1) The classification of the current Mineral Resource Estimate into Indicated and Inferred is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.

(2) All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.

(3) *All Resources are presented undiluted and in situ, constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction.*

(4) *Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*

(5) *It is envisioned that parts of the Denison deposit may be mined using open pit mining methods. In-pit mineral resources are reported at a cut-off grade of 0.3 % NiEq within a conceptual pit shell.*

(6) *The results from the pit optimization are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the Property. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.*

(7) *Underground (below-pit) Mineral Resources are estimated from the bottom of the pit and are reported at a base case cut-off grade of 1.1 % NiEq. The underground Mineral Resource grade blocks were quantified above the base case cut-off grade, below the constraining pit shell and within the constraining mineralized wireframes. At this base case cut-off grade the deposit shows good deposit continuity with limited orphaned blocks. Any orphaned blocks are connected within the models by lower grade blocks.*

(8) *Based on the size, shape, location and orientation of the Denison deposit, it is envisioned that the deposit may be mined using longhole open stoping (a bulk mining method that has long been utilized in the Sudbury region).*

(9) *High grade capping was done on 10 ft (3.05 m) composite data.*

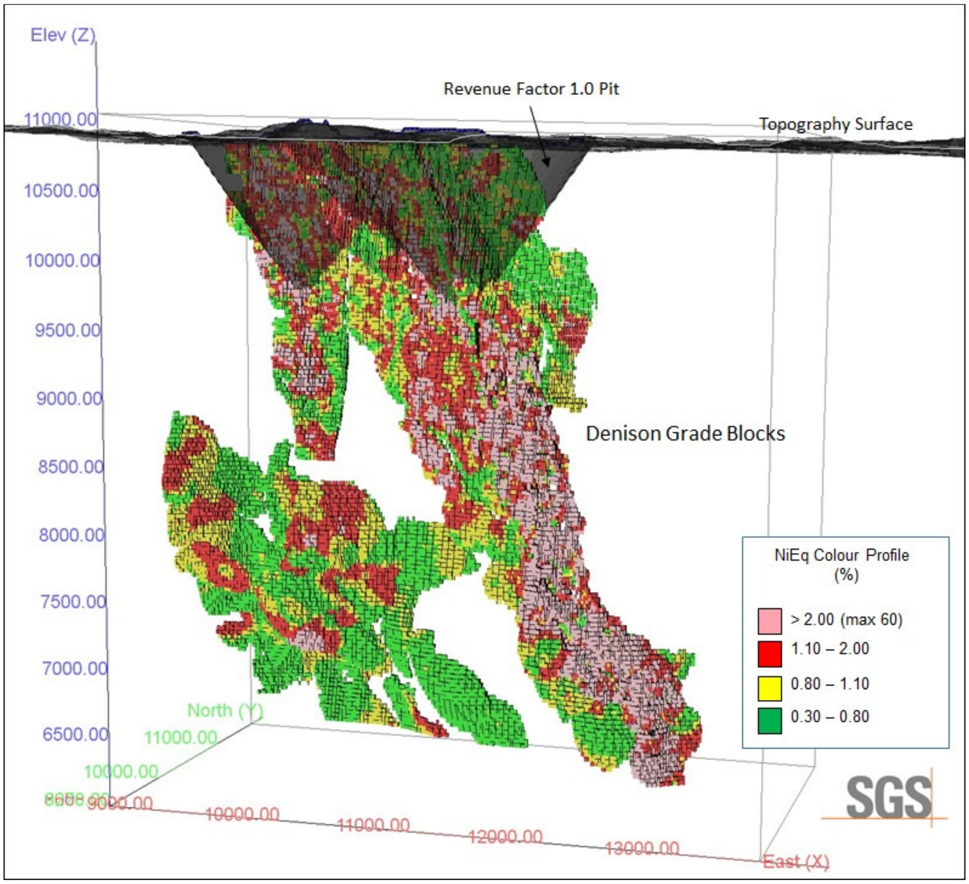
(10) *Bulk density values were determined based on physical test work from each deposit model and waste model.*

(11) *NiEq Cut-off grades are based on metal prices of $8.50/lb Ni, $3.752/lb Cu, $22.00/lb Co, $1000/oz Pt, $2000/oz Pd and $1,750/oz Au and metal recoveries of 78% for Ni, 95.5% for copper, 56% for Co, 69.2% for Pt, 68% for Pd and 67.7% for Au.*

(12) *The in-pit base case cut-off grade of 0.3% NiEq considers a mining cost of US$2.50/t rock and processing, treatment and refining, transportation and G&A cost of US$38.00/t mineralized material, and an overall pit slope of 55 degrees. The below-pit base case cut-off grade of 1.1 % NiEq considers a mining cost of US$80.00/t rock and processing, treatment and refining, transportation and G&A cost of US$42.50/t mineralized material.*

(13) *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*

Figure 14-8: Isometric View Looking North of the Crean Hill Deposit Mineral Resource Block Grades and Whittle Pit (upper and lower image)



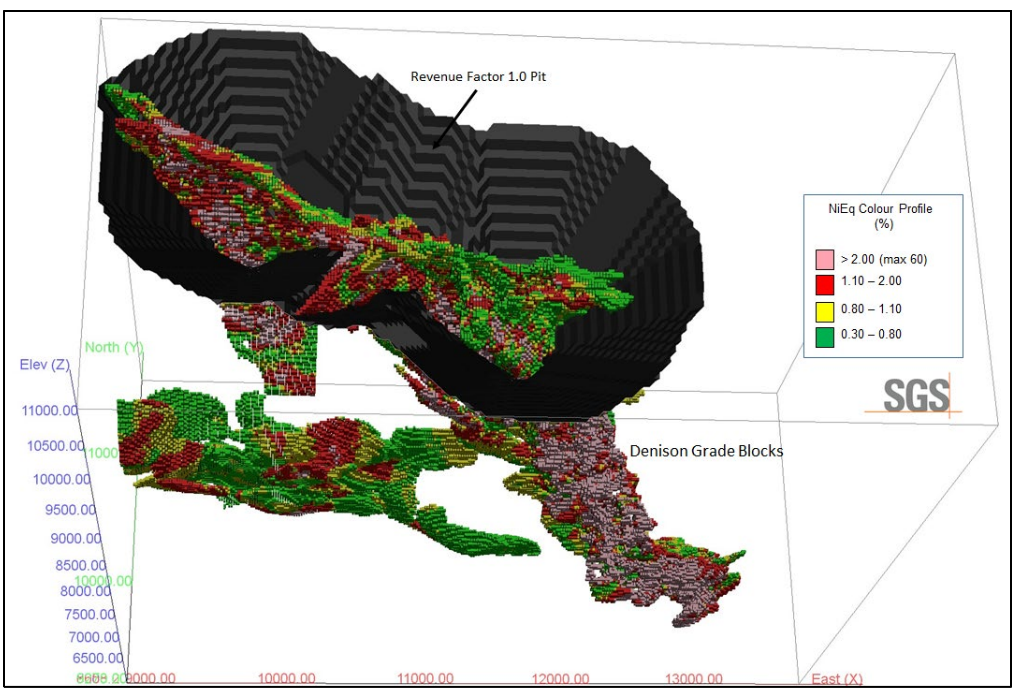
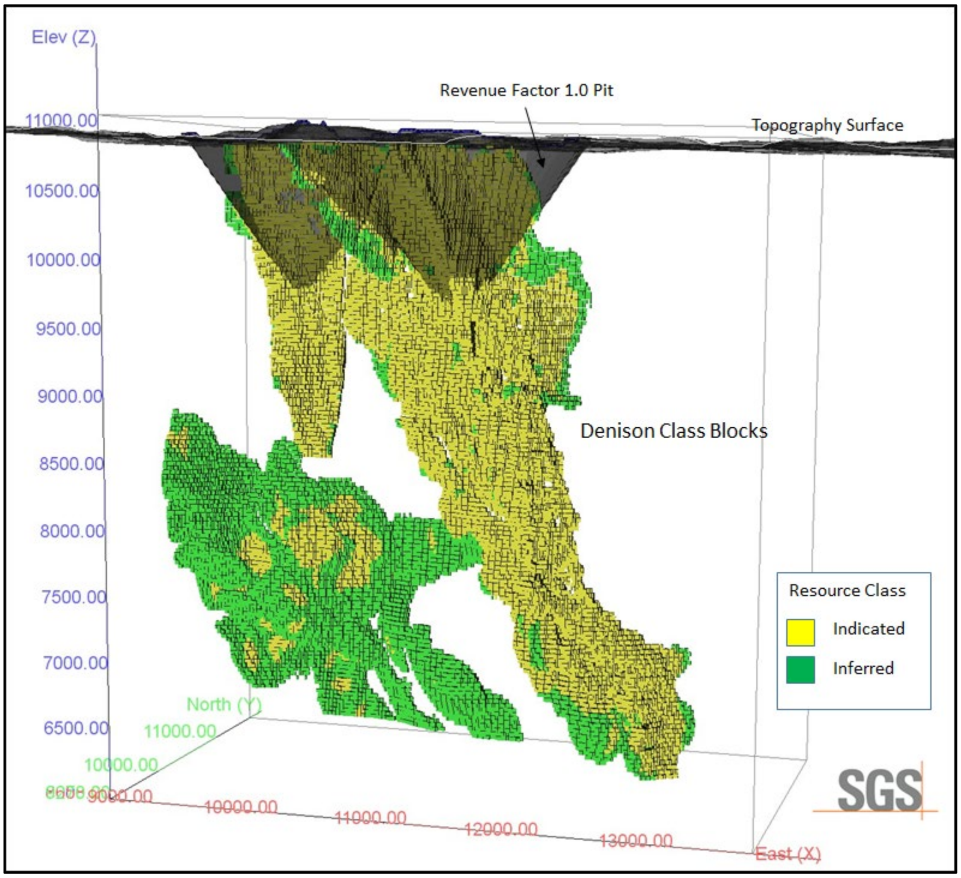
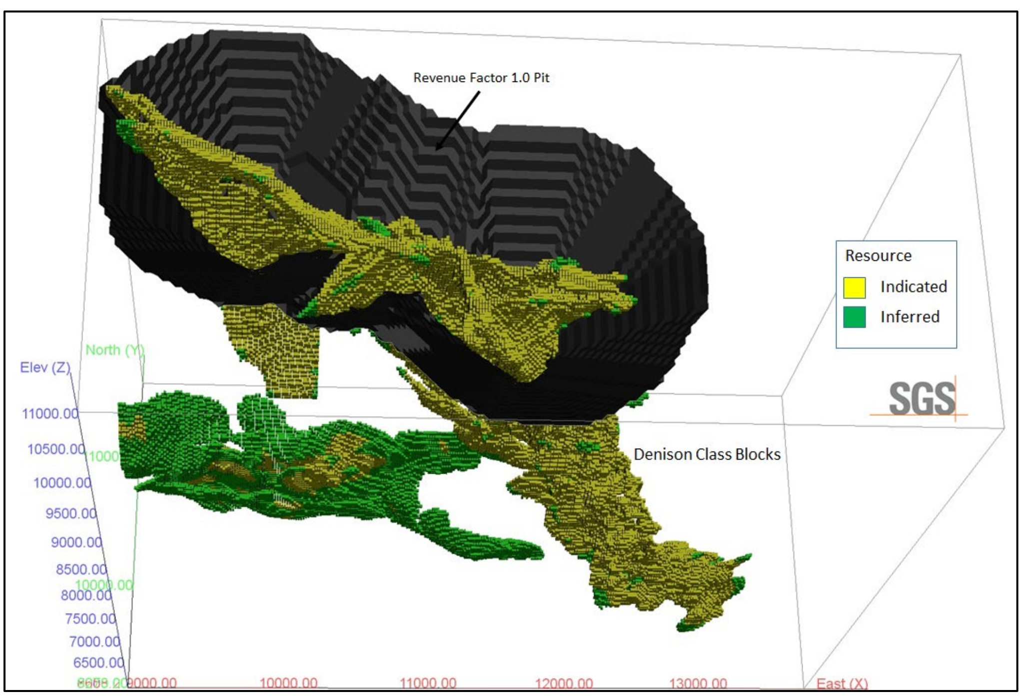


Figure 14-9: Isometric View Looking North of the Crean Hill Deposit Indicated and Inferred Mineral Class Blocks and Whittle Pit (upper and lower image)





14.11 MODEL VALIDATION AND SENSITIVITY ANALYSIS

The total volume of the Crean Hill Deposit resource blocks in the Mineral Resource model, at a 0.0% NiEq cut-off grade value compared well to the total volume of the 3D models, with the total volume of the block model being 3.44% lower than the total volume of the mineralized domains (Table 14-9). The slightly higher volume of the domains is the result of minor overlapping of domains not being counted in the MRE, and of limiting the search radius. Parts of the 99 Zone models were beyond the search radius. Where solids overlap, GEMS assigns the data to the first possible solid based on the "Solid Precedence" setting.

Visual checks of block grades against the composite data on vertical section showed good correlation between block grades and drill hole assay intersections.

A comparison of the average composite grades with the average grades of all the blocks in the block model at a 0.0% NiEq cut-off grade was completed and is presented in Table 14-10. The block model average grades compared well with the composite average grades. The lower block grades for Ni and Cu are likely due to grade smoothing during the interpolation procedure. The higher grades of precious metals in blocks are likely the result of ignoring unsampled precious metals during the interpolation procedure.

Table 14-9: Comparison of Block Model Volume with the Total Volume of the Deposit 3D Models (Before Removing Mined-Out Material)

Deposit	Total Domain Volume		Block Model Volume		Difference %
Denison Deposit	894,140,000 ft^3	25,319,000 m^3	863,355,000 ft^3	24,447,000 m^3	3.44%

Table 14-10: Comparison of Average Composite Grades (Based on Assayed Data) with Block Model Grades

Deposit	Variable	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
Denison Deposit	Composites Capped	0.72	0.66	0.03	0.38	0.36	0.14
	Blocks	0.60	0.53	0.02	0.53	0.54	0.30

14.11.1 Sensitivity to Cut-off Grade

The Crean Hill deposit Mineral Resource has been estimated at a range of cut-off grades to demonstrate the sensitivity of the MRE to cut-off grade (Table 14-11). Cut-off grades used for the MRE within a conceptual pit shell (0.3% NiEq) and below the conceptual pit shell (1.1% NiEq) are highlighted in the table.

Table 14-11: Crean Hill Deposit Open Pit (A) and Underground (B) Mineral Resource Estimate, July 4, 2022 at Various NiEq Cut-off Grades

(A)

Cut-off Grade NiEq (%)	Tonnes	Ni%	Cu%	Co%	Pt g/t	Pd g/t	Au g/t	NiEq%
Indicated								
0.2	17,241,000	0.52	0.48	0.02	0.47	0.36	0.25	1.06
0.3	16,760,000	0.53	0.49	0.02	0.48	0.37	0.25	1.08
0.4	16,080,000	0.55	0.50	0.02	0.49	0.38	0.26	1.11
0.5	14,977,000	0.57	0.52	0.02	0.50	0.39	0.27	1.16
0.6	13,528,000	0.61	0.55	0.02	0.53	0.42	0.28	1.22
0.8	9,961,000	0.70	0.62	0.02	0.62	0.50	0.32	1.41
Inferred								
0.2	440,000	0.43	0.48	0.02	0.28	0.14	0.07	0.81
0.3	434,000	0.43	0.49	0.02	0.29	0.14	0.07	0.82
0.4	410,000	0.45	0.51	0.02	0.30	0.15	0.08	0.84
0.5	326,000	0.49	0.58	0.02	0.29	0.19	0.10	0.94
0.6	283,000	0.53	0.62	0.02	0.33	0.22	0.11	1.00
0.8	192,000	0.61	0.70	0.02	0.32	0.16	0.16	1.14

(B)

Cut-off Grade NiEq (%)	Tonnes	Ni%	Cu%	Co%	Pt g/t	Pd g/t	Au g/t	NiEq%
Indicated								
0.8	21,678,000	0.78	0.70	0.03	0.73	0.82	0.45	1.70
1.0	16,789,000	0.89	0.79	0.03	0.82	0.94	0.51	1.94
1.1	14,531,000	0.96	0.84	0.03	0.88	1.02	0.54	2.07
1.2	12,581,000	1.02	0.90	0.03	0.94	1.10	0.58	2.22
1.3	10,909,000	1.09	0.95	0.04	1.01	1.18	0.61	2.37
Inferred								
0.8	4,039,000	0.50	0.41	0.02	0.44	0.64	0.15	1.07
1.0	1,779,000	0.58	0.47	0.02	0.56	0.89	0.19	1.29
1.1	1,170,000	0.61	0.46	0.02	0.64	1.09	0.21	1.41
1.2	754,000	0.67	0.50	0.02	0.62	1.32	0.21	1.56
1.3	539,000	0.73	0.51	0.02	0.63	1.44	0.23	1.68

(1) In-pit Mineral Resources are reported at a base case cut-off grade of 0.3% NiEq within a conceptual pit shell and underground (below-pit) Mineral Resources are reported at a base case cut-off grade of 1.1% NiEq from the bottom of the conceptual pit shell. Values in this table reported above and below the base case cut-off grades should not be misconstrued with a Mineral Resource Statement. The values are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. All values are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.

(2) *All figures are rounded to reflect the relative accuracy of the estimate. Composites have been capped where appropriate.*

14.12 DISCLOSURE

All relevant data and information regarding the Project are included in other sections of this Technical Report. There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading.

The Author is not aware of any known mining, processing, metallurgical, environmental, infrastructure, economic, permitting, legal, title, taxation, socio-political, or marketing issues, or any other relevant factors not reported in this technical report, that could materially affect the current Mineral Resource Estimate.

15 MINERAL RESERVE ESTIMATES

No mineral reserves are being disclosed.

16 MINING METHODS

16.1 INTRODUCTION AND GEOLOGICAL SETTING

As described in detail in Section 7 – Geologic Setting and Mineralization, the nickel, copper, and platinum group element (Ni-Cu-PGE) deposits in Sudbury are theorized to have formed as a result of a major early Proterozoic meteorite impact 1,850 million years ago (Ames and Farrow, 2007), (SGS Technical Report, 2022). This geological event resulted in what is today known as the Sudbury Igneous Complex (SIC), a mining camp with a documented mining history going back to the late 1800s. Today, Sudbury is a world-renowned source of refined nickel and other metal products.

The SIC is geographically divided into the North, South, and East Ranges. The Crean Hill property (which hosts the Crean Hill deposit) is located within the South Range of the SIC.

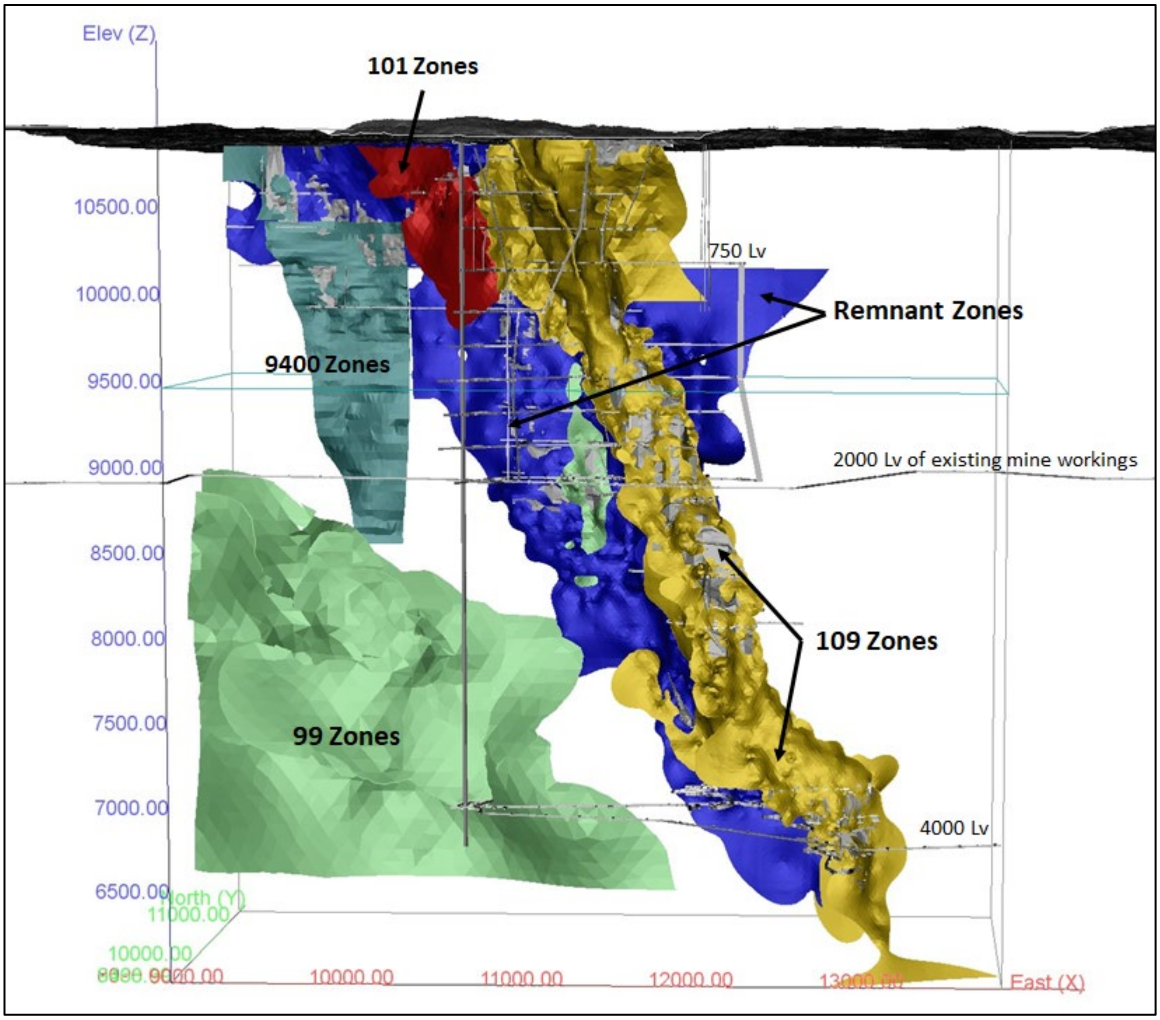
The Crean Hill property is part of a large trough structure at the base of the SIC. This trough structure contained several previously mined ore deposits including the Crean Hill Main Orebody, the Crean Hill Intermediate Orebody, the Crean Hill West Orebody, Ellen Mine, and Lockerby Mine, each sitting in embayments (terraces) within the larger trough. Much of the mined Ni-Cu contact mineralization is associated with the embayment structures in the SIC, and the embayments largely control the distribution of Ni-Cu mineralization. It is believed that additional embayments in the SIC containing significant Ni-Cu sulphide mineralization may be present at Crean Hill.

The strike of the SIC contact ranges from 120° at surface to 80° at depth, and the dip varies from steeply dipping to the north at surface through vertical to steeply dipping over-turned to south at the lower depths. The contact between the SIC and the footwall is frequently sheared. Shearing and brittle faulting also occur within the footwall.

There is one main fault in the immediate area of the 109 FW zone, a shallow fault striking 100° and dipping 25° south. This fault is comprised of two or more horizons, where core is broken up along poorly healed joints.

Figure 16-1 shows an isometric view of the orebody with the different zones as they have been delineated and named over time.

Figure 16-1: Isometric View Looking North: Crean Hill Deposit Models (grid units are feet)



16.1.1 Mining Methods Selection

The Crean Hill Ni-Cu-PGE deposit occurs from relatively shallow depths but has been explored and intermittently mined to depths of more than 4,000 ft below surface.

Due to the steeply dipping nature of the deposit and the known extent at depth, underground mining is the most appropriate method to extract most of the mineralized resource, and the selected underground mining methods, stope optimization, and mine design will be discussed in Chapter 16.3 Underground Mining.

However, the presence of mineralized outcrops has been known for a long time, and just over a million tonnes of ore has been mined using open pit mining methods in the past.

For this reason, both open pit and underground mining methods were considered for the evaluation and development of the Crean Hill Project.

16.2 OPEN PIT MINING

Since the open pit and underground mine would likely be connected owing to continuous mineralization, leading to a combined design, an iterative process was followed. The first step in the process was to determine the most likely size or expected ultimate pit shell for the open pit portion of the mine.

All open pit evaluation work for Crean Hill, including pit optimization and pit design, was based on the 2022 geological block model developed by SGS. The characteristics of this block model and the mineral resource estimation process that was followed to build it is described in Section 14 – Mineral Resource Estimates.

16.2.1 Pit Geotechnical Design Parameters

No new geotechnical drilling data or information was available for this evaluation of the Crean Hill open pit. Similar to the 2020 PEA by SRK Consulting (Canada), geotechnical input was based off geotechnical recommendations completed by Tetra Tech in a 2012 study.

For the 2020 PEA, SRK assumed steeper slope angles than those recommended by Tetra Tech. For this 2023 open pit evaluation, Stantec opted for a more conservative approach and assumed slope angles that closely resemble Tetra Tech's.

Table 16-1 shows the slope design sectors and design angles as recommended by Tetra Tech.

Table 16-1: Tetra Tech Slope Design Recommendations from 2012

Geotechnical Domain	Wall Sector Azimuth *(degrees)*	Bench Height *(m)*	Catch Bench Width *(m)*	Bench Face Angle *(degrees)*	Design Inter Ramp Angle *(degrees)*
Domain I – N (Norite)	330 – 60	22.8	10.7	60	32
Domain I – N (Backfill)	330 – 60	7.6	6.1	37	28
Domain II – SE (MTBS)	60 – 120 120 – 180	22.8 22.8	10.7 11.3	65 70	46 26
Domain III – SW (MTBS)	180 – 210 210 – 330	22.8 22.8	11.3 10.7	70 65	26 51
Domain III – SW (Diabase Dyke)	150 – 300	22.8	12.2	65	33

Source: Tetra Tech, 2012

Note that the Tetra Tech values do not specify overall slope angles (OSAs), as these would be dependent on the depth of the pit and location of ramps.

For pit optimization purposes, Stantec simplified these recommendations, but kept the assumed values close to the more conservative Tetra Tech recommended values. Table 16-2 shows the overall pit slope angles assumed for this study.

Table 16-2: Overall Pit Slope Angles Assumed for this Study

Geotechnical Domain	Wall Sector Azimuth (°)	Bench Height (m)	Bench Face Angles (°)	Design Overall Slope Angle (OSA) (°)
All Domains – (Norite & MTBS)	0–360	5.0	70	38
Old Workings Backfill Materials	0–360	5.0	37	27

16.2.2 Economic Pit Optimization Methodology

The economic pit optimization process for the Crean Hill open pit had to be performed in two stages. The reason for this was that waste rock would be stored in two separate locations, one close to the proposed pit and one several kilometers away. The two-stage pit optimization process would allow for the application of two mining unit costs, based on the different haulage costs.

Waste material would be stored nearby on the property for the first stage of the optimization. Given the surface restrictions for dump storage and the requirement to relocate legacy waste material, Stantec estimated that approximately 9.65 Mt of waste could be stored on-site prior to designing the facilities. This material estimate served as a restriction for the maximum amount of waste that could be extracted from the pit during the first stage of the optimization. To account for the cost of moving legacy waste material, an initial capital cost of $10.5 million (estimated by Magna) was applied to the optimization.

Both stages of the optimization were restricted by the following surface constraints.

- Crean Hill property boundary
- Shaft Pillar (for legacy mine workings)
- Estimated extents of the capped landfill facility
- Water dam north of pit (restriction is 100 ft away from the estimated toe of the dam)
- Reclamation boundary estimate

The optimal pit shell was determined by using the pseudo-flow algorithm as embedded in the GEOVIA Whittle^{TM} software. The input parameters for the pit optimization were obtained and agreed upon at the start of the design phase.

16.2.3 NSR Calculation

To account for revenue from mined blocks, a NSR value was calculated for every block in the block model, expressed as a value per short ton with a CAD$/Ton unit. An exchange rate of 1.30 CAD$/USD$ was assumed for calculating NSR values.

Based on the provided parameters in the commercial terms provided by the smelter, the main determinants of block value were the following.

- Metal price assumptions (as presented in Table 16-3)
- Processing recovery for each element
- Concentrator treatment, smelter treatment, and refining charges
- Smelter payability rates for each metal
- Transportation costs

Table 16-3: Metal Prices

Metal	Selling Price
Nickel	US$ 9.50/lbs.
Copper	US$ 3.50/lbs.
Cobalt	US$ 22.00/lbs.
Platinum	US$ 1,000/Troy Oz.
Palladium	US$ 1,800/Troy Oz.
Gold	US$ 1,700/Troy Oz.

Indicative commercial smelting terms from a Sudbury-based processing plant were provided to Stantec by Magna. The terms cannot be included in this report due to confidentiality restrictions. The NSR was structured to account for a bulk nickel and copper concentrate, as well as a separate copper concentrate product. Cobalt, platinum, palladium, and gold also contribute value to the concentrate products according to the structure of the smelter's commercial terms.

NSR values were calculated on a block-by-block basis. The structure of the calculation was verified by Magna's processing consultant, XPS. Metal grades from the block model were used to calculate NSR values on a CAD$/Ton basis. The NSR values were then loaded into HxGn MinePlan™ and Deswik for open pit and underground mine planning, respectively. For open pit mine planning, NSR values were exported from HxGn MinePlan™ to GEOVIA Whittle™ as part of the block model, which also contained data regarding density, and geotechnical domains defined by Stantec.

16.2.4 Pit Optimization

Utilizing the NSR values in the block model, the Whittle™ pit optimization software was set up, and several optimization runs were conducted.

The processing cut-off value was determined by the processing cost. On an undiluted basis, the processing cost build-up for the open pit (and the cut-off value) is presented in Table 16-4. Note that the block model was created on a US Tons basis. Table 16-5 outlines the parameters used for the first stage of the optimization. A stockpiling strategy has not been incorporated into the optimization process at this time.

Table 16-4: Cut-off Value for Open Pit Processing

Item	Value
Processing Cost	CAD$ 35.38/Ton
Crushing and loading cost (for off-site transportation)	CAD$ 2.00/Ton
Transport cost from site to mill (off-site)	CAD$ 5.00/Ton
Cut-off value for processing (Total)	CAD$ 42.38/Ton

Table 16-5: Pit Optimization Input Parameters for the First Stage of the Optimization

Parameter	Value
Mining Cost	$CAD 5.69/Ton
Processing Cost	$CAD 42.38/Ton
Overall Pit Slopes Through in-situ Rock	38°
Overall Pit Slopes Through Backfill	27°
Dilution	5%
Mining Recovery	95%
Discount Factor	8% per year
Mining Rate Limit	7.0 MTpa
Processing Rate Limit*	1.08 MTpa
Revenue Factor increments	0.01

*Equivalent to 3,000 Tpd at 360 operating days per year.

Note that revenue factors were applied as increments of the selling price of metals. For example, a pit created at revenue factor 0.50 would be created with 50% of the NSR value of a block, but with the full mining and processing costs applied. Creating pit shells in revenue factor increments results in a series of nested pit shells that can be used to guide pit design and sequencing.

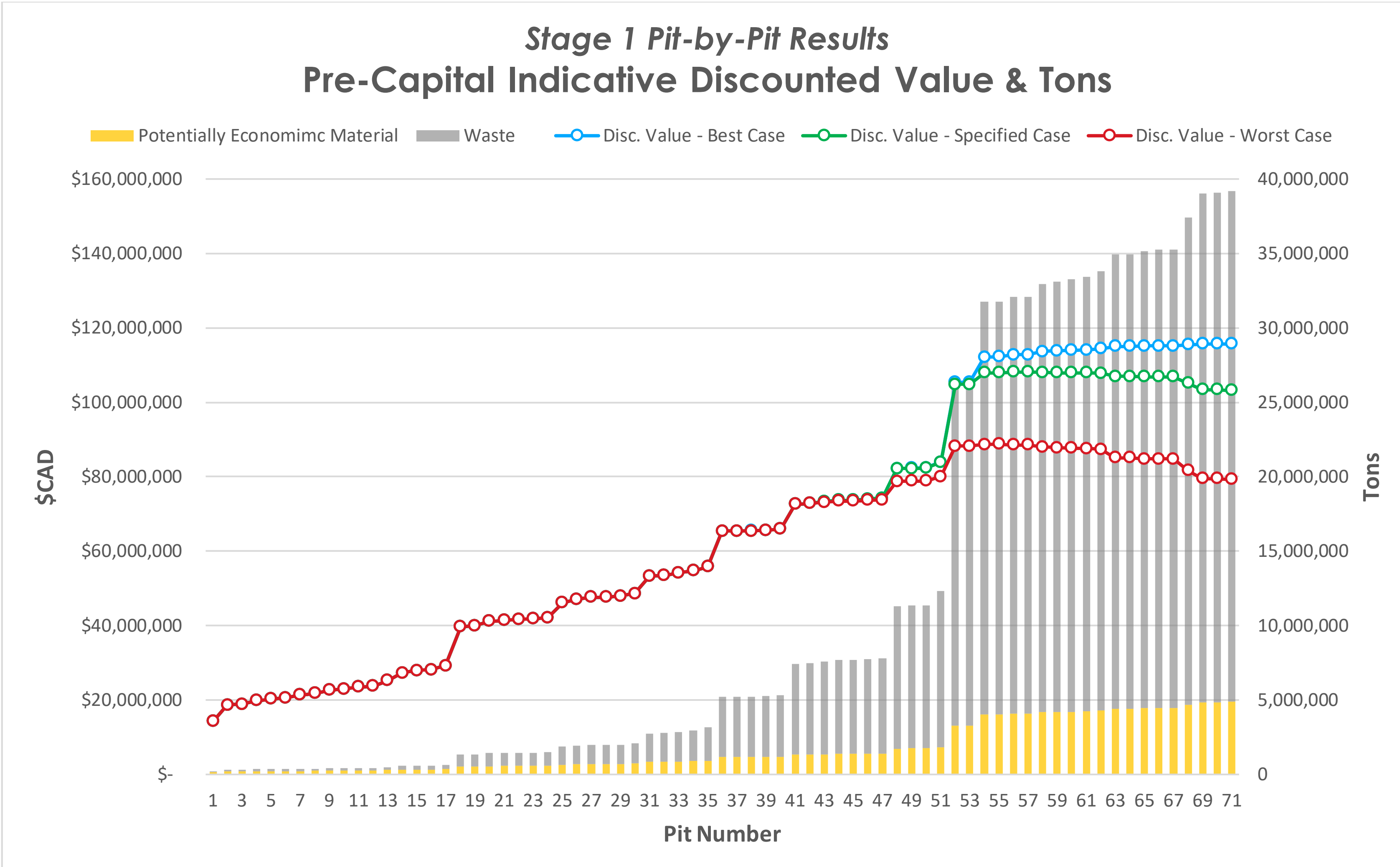
The typical output from the WhittleTM software is a pit-by-pit graph that provides a graphical representation of the incremental values provided by each pit shell, and the associated pit tonnages. Figure 16-2 shows the results of the first pit optimization run in WhittleTM. The secondary vertical axis shows the tons of material contained in the respective pits, and the primary vertical axis shows the discounted value of the respective pit shells. Discounted values presented are indicative only and do not take capital, royalties, taxes, or any other costs into consideration other than the items presented above.

The discounted value for each pit is calculated for the best, the specified, and the worst-case scenario. All three scenarios are described as follows.

- **Best case**: Discounted value is determined by scheduling every preceding pit as a pushback to create as many mining phases as possible.
- **Specified case**: Discounted value is determined by scheduling the pit with selected pit phases.
- **Worst case**: Discounted value is determined by mining the ultimate pit without any mining phases.

The specified case mines Pit 36 (revenue factor 0.65) and Pit 48 (revenue factor 0.77) as interim pushbacks to increase the estimated discounted value during the pit optimization process. Interim pushbacks were selected based on the value added and the practicality of mining them. The actual mining sequence will be different, but it will be guided by the phases produced from the pit optimization. In this case, Pit 48 was one of the largest practical pushbacks that could be mined within the waste mining limit, and Pit 48 therefore serves as the ultimate pit for the first stage of the optimization. The results of the first stage are shown in Figure 16-2.

Figure 16-2: Stage 1 Pit Optimization Results



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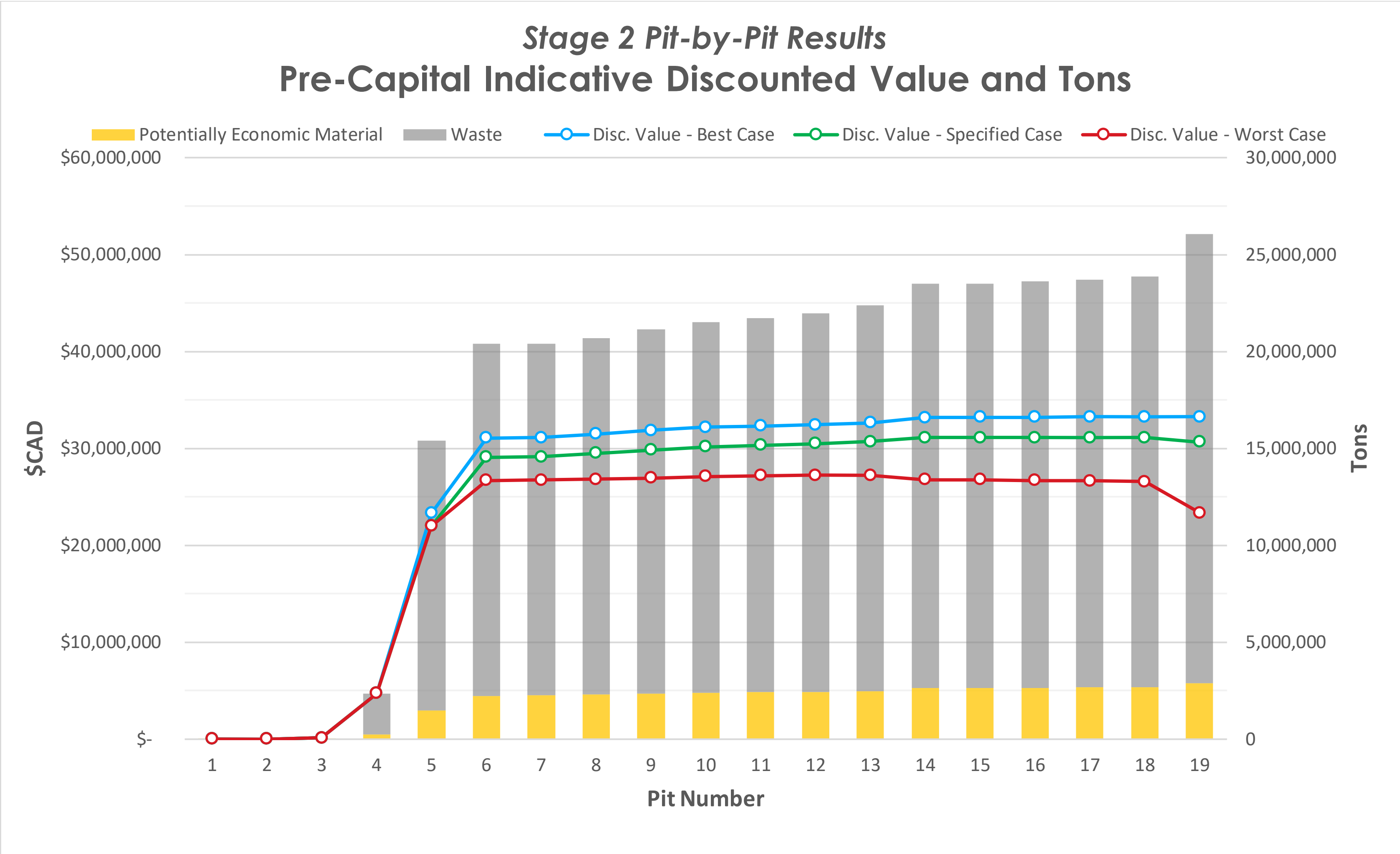
The second stage of the optimization was similar to the first stage, except the material within Pit 48 was depleted from the model (to avoid double counting), and waste material was hauled off-site to the nearby Ellen Pit (discussed further in Section 16.2.4). As such, the cost of mining waste was increased to account for hauling material off-site. The incremental waste haulage cost was estimated to be approximately CAD$ 0.37/Ton, which increases the waste mining cost to CAD$ 6.06/Ton. This was estimated using Stantec's internal costing database. Incremental haul cycle times were estimated using Talpac software. The ore mining cost is unaffected by this change. The only other change made was increasing the mining rate limit to 10 MTpa to account for the higher strip ratio.

The results of the second stage of the optimization are presented in Figure 16-3.

Figure 16-3: Stage 2 Pit Optimization Results



As before, discounted values shown are indicative and only consider the cost items discussed in this section. These indicative discounted value estimates have been discounted at a rate of 8% per year for 1.9 years to account for time lapsed during mining of the first stage of the optimization. These indicative discounted values are incremental to the first stage of the optimization.

This second stage of pit optimization resulted in two additional phases to the pit, which were used to develop the Crean Hill design pit. Stantec selected Pit 5 (revenue factor 0.84) and Pit 19 (revenue factor 1.00) on the basis that they form practical pushbacks and maximize the potential of the pit, within the bounds of the metal prices assumed.

In general, pit shells produced cannot be mined exactly as defined during pit optimization, because the geometry must be adjusted to accommodate mining equipment. The pit design process tends to smooth the pit geometry so that the pit still follows the general shape of the pit shell, but design pits tend to capture more waste, and slightly less ore.

16.2.5 Mining Equipment

Two design criteria were considered in the selection of the fleet. These were the following.

- Drilling, blasting, loading, and hauling up to 41,000 tons of material per day out of the pit.
- Haul trucks' ability to negotiate a 12% ramp in all weather conditions.

To satisfy the second requirement, an articulated dump truck (ADT) with a 45 short Ton payload was selected as the prime earth mover. Although these trucks have a higher hourly operating cost relative to rigid-frame trucks, they are all-wheel driven, able to maintain traction in muddy and wet conditions, and more capable of negotiating steep ramps.

The narrow width of these trucks is important for the pit design, allowing for ramp widths of 50 ft for dual-lane roads. For reference, 45-Ton CAT 745 ADTs are 12.5 ft wide (3.8 m), whereas 42-Ton CAT 770G rigid frame trucks are 15.6 ft wide (4.8 m). As mentioned in section 16.2.2, it was important to reduce the ramp width to a minimum to access as much mineralized resource as possible while keeping the strip ratio to a minimum.

Since the 45-Ton ADT is smaller than the typical mining truck, there is a smaller selection of loaders and excavators available for good pass matching with the selected hauler. One excavator that comes with a bucket size suitable for a 3-pass match to the 45-Ton ADT is the CAT 390DL. This excavator can load the 45-Ton ADT in 3 passes and was selected for the two primary loading units. The primary loader will be complemented by a CAT 988H front-end loader (FEL) that will be able to quickly travel between working areas, as needed.

The specific equipment models selected are examples of the size and types of machines Stantec would select for this project. This selection is neither an endorsement nor a recommendation by Stantec of this manufacturer, or these models. Similar units from other manufacturers may be more suitable.

The Crean Hill open pit mine is intended to be a contractor-run operation. Magna's level of involvement in the equipment procurement process will depend on the arrangement negotiated with the site contractor.

The primary equipment fleet in this section was selected to provide a design basis for the pit design geometry, and production rates for the mine schedule.

To round out the primary equipment fleet, the minimum following equipment would also be required.

- One FEL for loading crushed product to highway trucks
- Two track dozers (CAT D8 size) – one in the pit and one at the rock storage facility
- One medium size road grader
- Three drill rigs
- One rubber-tire-dozer – for cleaning loading areas and road maintenance

A number of smaller support equipment would also be required to operate the open pit safely and efficiently. This equipment would include at a minimum the following.

- Small excavator for sump development and maintenance
- Water truck for dust suppression
- Diesel bowser or fuel truck for in-pit refueling.

16.2.6 Pit Design

The Crean Hill open pit was designed to extract mineralized material from the steeply dipping nickel-copper resource of the SIC. The steeply dipping nature of mineralization meant that extracting ore at depth was only possible by mining a substantial amount of waste material. The pit design was spatially limited by surrounding surface constraints (an underground shaft pillar, a landfill, and a water dam) and needed to access ore approximately 450 ft below surface. The target depth, surface constraints, and high strip ratio of the deposit required the pit design to limit the use of highwall ramps and use inter-phase ramps wherever possible.

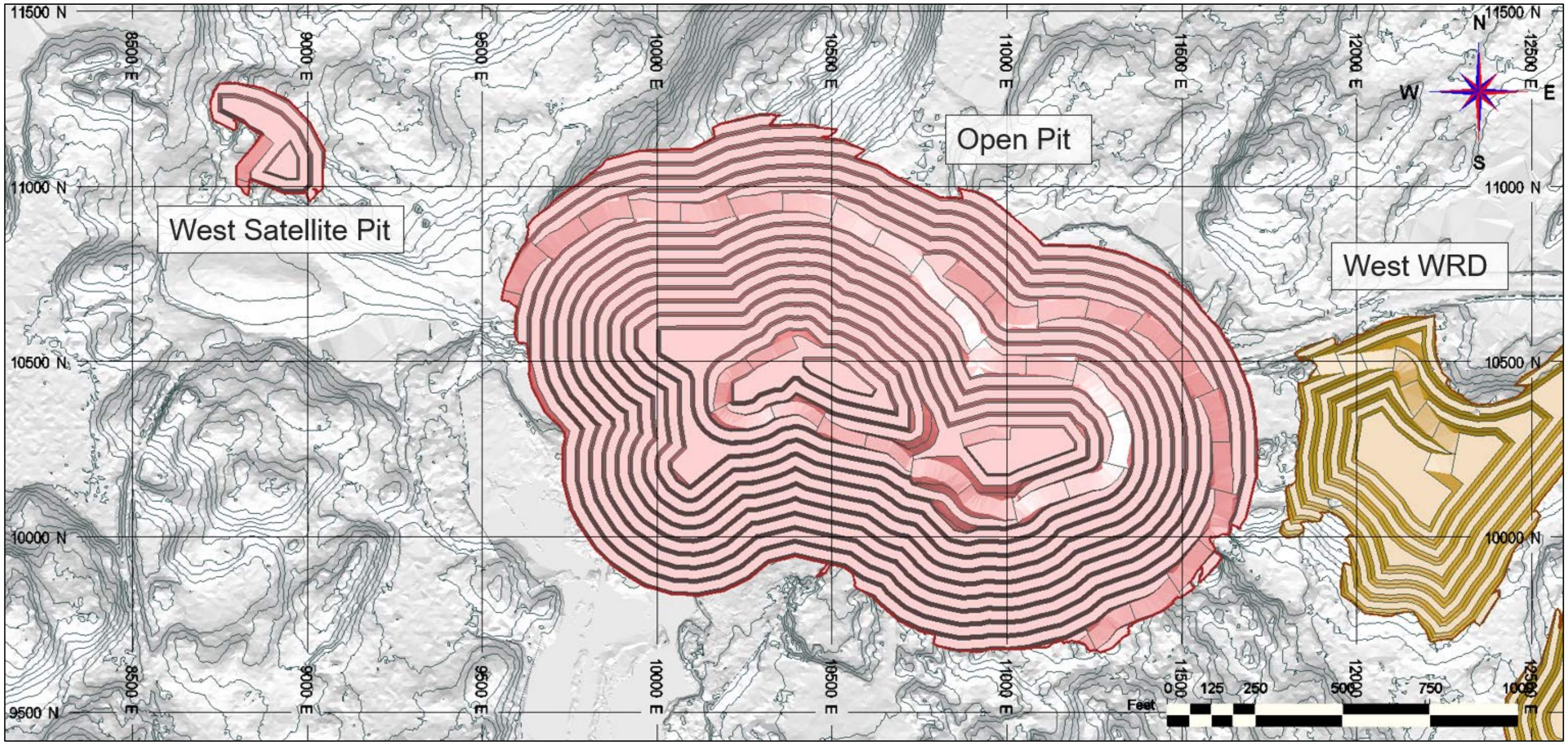
Incorporating addition highwall ramps increases the volume of waste material to be mined. The Crean Hill deposit has fixed surface constraints which meant that adding more highwall ramps would not substantially change the waste volume, but it would significantly reduce the amount of high-value ore recovered from the bottom of the pit. Using the pit design criteria defined in Table 16-6, Stantec refined the pit design through a series of iterations to create a robust access plan. The pit design plan is shown in Figure 16-4.

Table 16-6: Pit Design Criteria

Item	Metric	Imperial
Overall Pit Slopes Through Bedrock	38°	
Overall Pit Slopes Through Backfilled Material	26.6°	
Bench Face Angle Through Bedrock	70°	
Bench Face Angle Through Backfilled Material	37°	
Designed Bench Height	10 m	32.80 ft

Item	Metric	Imperial
Working Bench Height	5 m	16.40 ft
Catch Bench Width	8 m	26.20 ft
Dual Haul Road Width	15.25 m	50 ft
Single Lane Haul Road Width	10.7 m	35 ft
Maximum Ramp Grade	12%	

Figure 16-4: Crean Hill Open Pit Ultimate Design



The pit was designed with 32.8 ft (10 m) high benches and is intended to be mined in 16.4 ft (5 m) high double benches. The split bench configuration was chosen to support mining with smaller equipment and to allow for wider catch benches to reduce the risk to personnel from falling rock without sacrificing ore recovery. A 16.4 ft (5 m) deep sub-level excavation was planned to maximize pit-bottom ore recovery. Since there was not sufficient space for a ramp, this sub-level excavation was designed to be excavated from the bench above to maximize ore recovery down to the 10,513.2 ft level.

No data regarding overburden was made available but based on site visits and knowledge of the area, overburden is believed by Stantec to be relatively thin in this area and is not expected to significantly influence pit slopes. Additional geotechnical investigations are required to confirm the geotechnical parameters presented.

The pit was designed to contribute 3.83 million tonnes of potentially economic material, as shown in Table 16-7. Cross sections show potentially economical material if it is classified as a measured, indicated, or inferred resource. Approximately 98% of the potentially economic material is classified as indicated, and the remaining 2% is classified as inferred.

Table 16-7: Crean Hill Open Pit Material Quantities and Grades

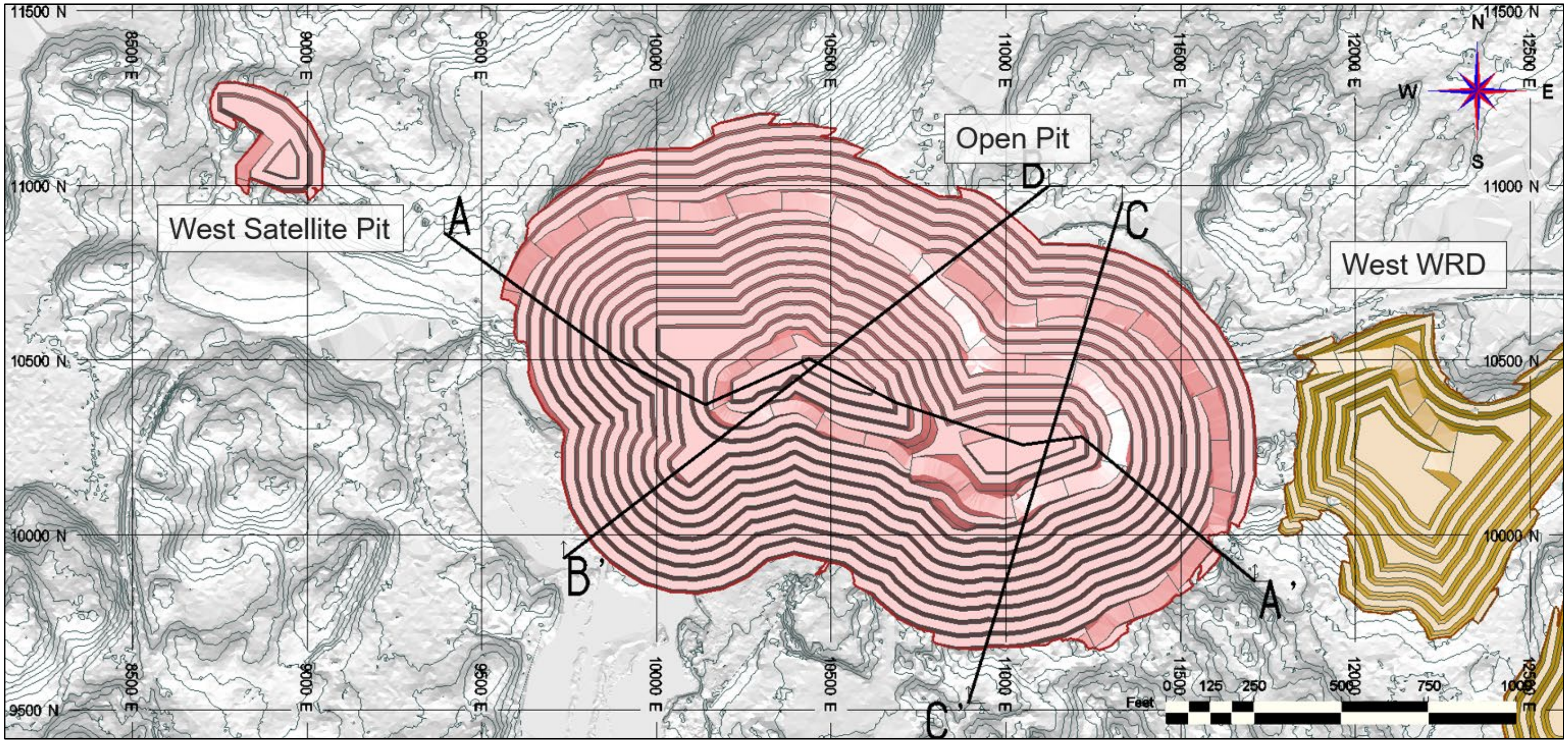
Material Tonnages	**Value**	**Units**
Potentially Economic Material	3.83	Million tonnes
Legacy Waste	4.28	Million tonnes
Pit Waste	29.35	Million tonnes
Total	33.18	Million tonnes
Strip Ratio	8.78	:1
Grades		
Dilute NSR Value	131.05	CAD$/tonne
Dilute Nickel Grade	0.53	%
Dilute Copper Grade	0.41	%
Dilute Cobalt Grade	0.019	%
Dilute Platinum Grade	0.36	ppm
Dilute Palladium Grade	0.22	ppm
Dilute Gold Grade	0.17	ppm

Notes:

1. Potentially economic material tonnage and grade values are diluted at 5%
2. A mining recovery rate of 95% was applied.
3. Ore tonnage were determined with a dilute NSR cut-off value of $42.38/Ton

Cross sections showing the pit design and estimated NSR values are presented in Figure 16-5 through Figure 16-8. The cross sections only show blocks from the model that are a classified resource.

Figure 16-5: Locations of Sections



The pit is shown with 16.4 ft (5 m) contours, topography is shown with 5 ft (1.5 m), and the west waste rock dump (WRD) is shown with 8.4 ft (2.5 m) contours.

Figure 16-6: Long Section AA' Showing Ultimate Pit Outline and Orebody Geometry

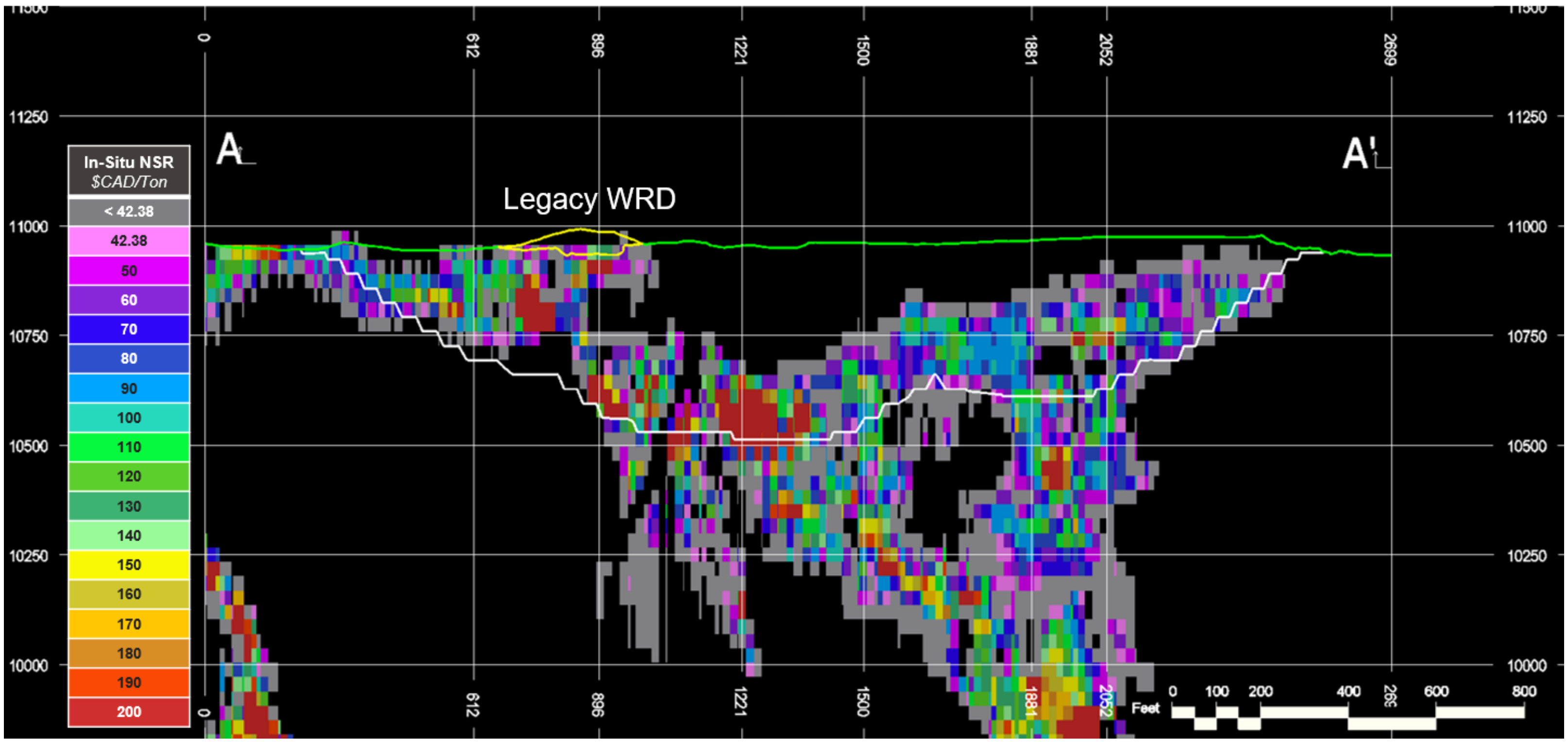


Figure 16-7: Cross section BB' showing the historic waste rock storage facility

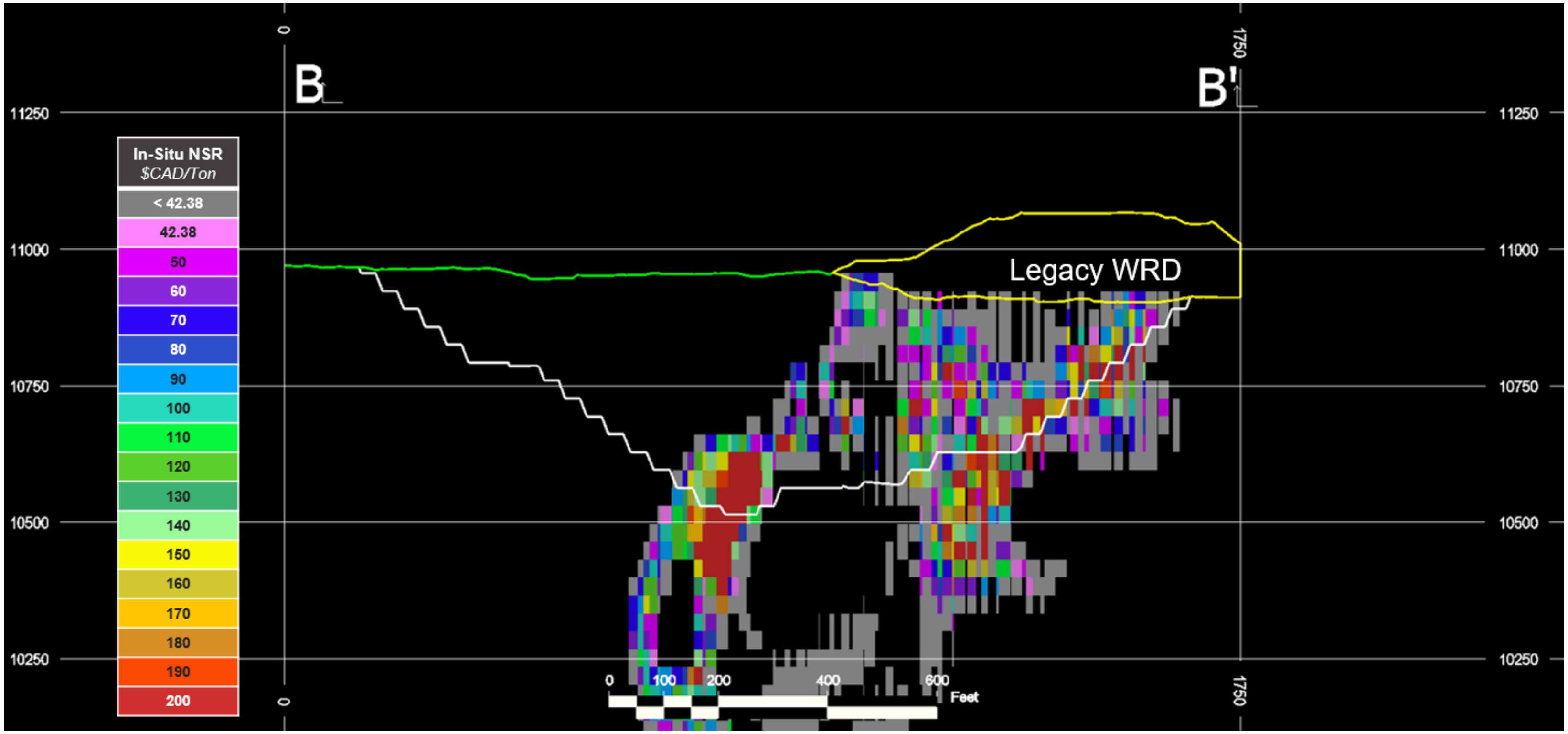
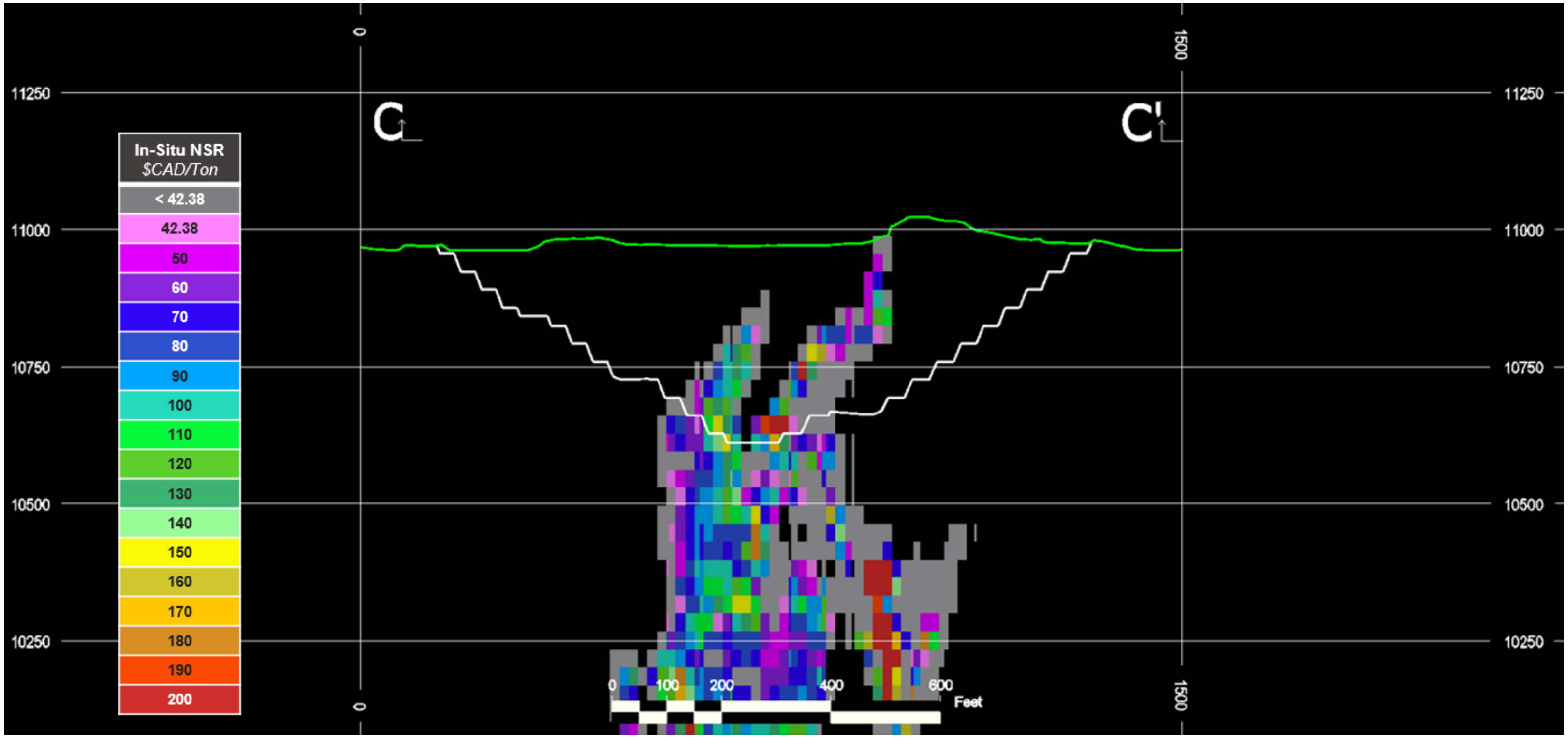


Figure 16-8: Cross section CC' through the east lobe of the main pit



A robust access plan for the pit was key to maximize ore recovery and support the development of the four mining phases. The pit was designed with two main access ramps to support mining down to the 10,792 ft level (the ramp merge point). The west ramp was developed clockwise, and the east ramp was developed counterclockwise. Both were developed at a relatively steep grade of 12%. While steeper than normal, they are well within the limits of ADTs. Using these ramps efficiently will require excellent road maintenance to handle rain, snow, and ice.

The west ramp provided access from pit crest to pit-bottom. The east ramp was required to access benches above the 10,792 ft level in the phase 4 pit, after the upper levels of phases 1-3 are mined (discussed further in Section 16.2.4). This ramp was designed for phase 4 specifically but could also be used during any period that an inter-phase ramp could be established to reach it. The main ramp was reduced from a dual-lane width to single-lane width below the 10,628 ft level to access additional high-grade ore near pit-bottom. Haul traffic congestion is not anticipated to be a significant issue given the single-lane ramp only travels from the 10,628 ft level to the 10,529.6 ft level, and these benches have a low strip ratio.

A portion of the existing access road through the property will have to be removed and reconstructed adjacent to the pit to support development of the surface mine. This will have to be addressed with the relevant stakeholders. The pit is shown with 16.4 ft (5 m) contours, topography is shown with 5 ft (1.5 m), and the west waste rock dump (WRD) is shown with 8.4 ft (2.5 m) contours.

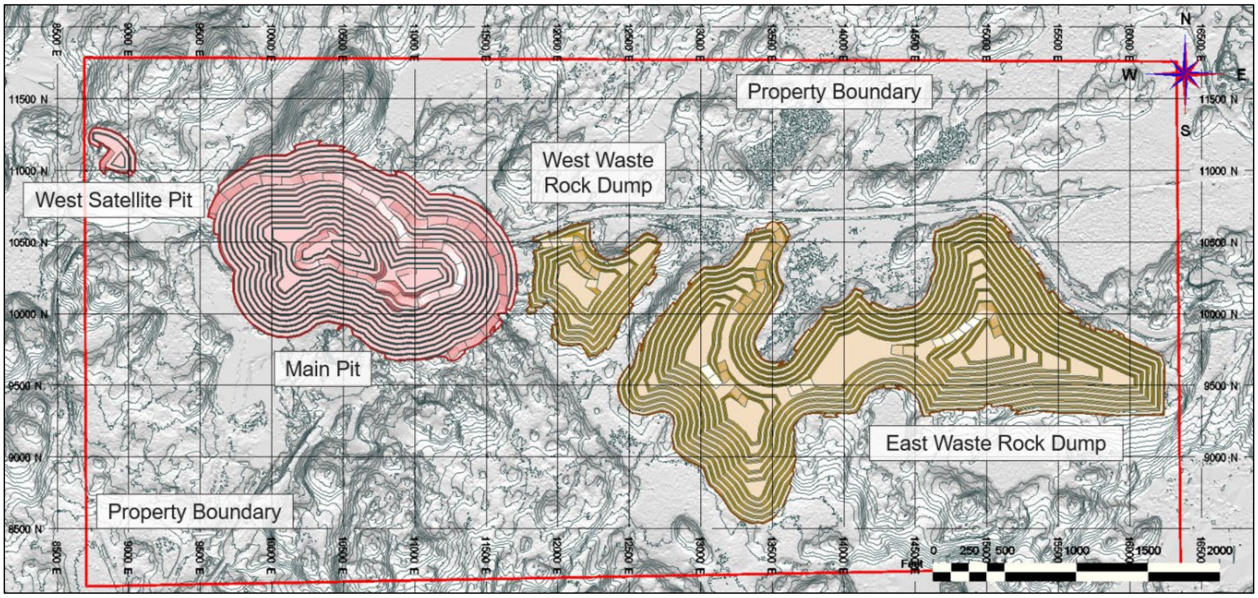
16.2.7 Waste Rock Storage

Stantec has identified two areas on the Crean Hill property that are suitable for waste rock storage on the basis that they are close enough to the pit and will not be built overtop of existing water features. The area south of the main pit was not selected for waste storage because it would interfere with the Monk Lake watershed, which drains to a water management system owned by a third party. Design criteria for the WRDs are as follows.

- Overall slope of 2.5:1 (Horizontal:Vertical)
- Lift height of 16.4 ft (5 m)
- Catch bench width of 19.3 ft (5.9 m)
- Minimum road width of 50 ft (15.2 m)
- Maximum ramp grade of 12%

The WRD configuration is shown as part of the site plan in Figure 16-9. The West WRD has a maximum elevation of 11,046.2 ft (3,367 m) and is approximately 75 ft (23 m) high. The East WRD has a maximum elevation of 11,152.8 ft (3,399 m) and has a maximum height of approximately 185 ft (57 m).

Figure 16-9: Crean Hill Site Plan



The legacy WRD situated overtop of the designed pit will also need to be relocated. Legacy waste material will be stored in the East and West WRD facilities. The East and West WRD facilities do not have sufficient storage capacity to store all waste material from the legacy WRD, as well as waste mined from the pit. Magna Mining Inc. has assumed that all surplus waste from the pit can be stored in the Ellen pit immediately east of the property. Table 16-8 summarizes waste rock production, and Table 16-9 summarizes the waste rock storage capacity required.

Table 16-8: Waste Production

Source	**Waste** (million tonnes)
Legacy Waste	4.3
Pit Waste	29.3
Total	33.6

Table 16-9: Waste Storage Capacity

Destination	**Waste** (million tonnes)
West WRD	0.8
East WRD	13.1
Total On-Site Storage Capacity	13.9
Off-Site Storage Capacity Required	19.7

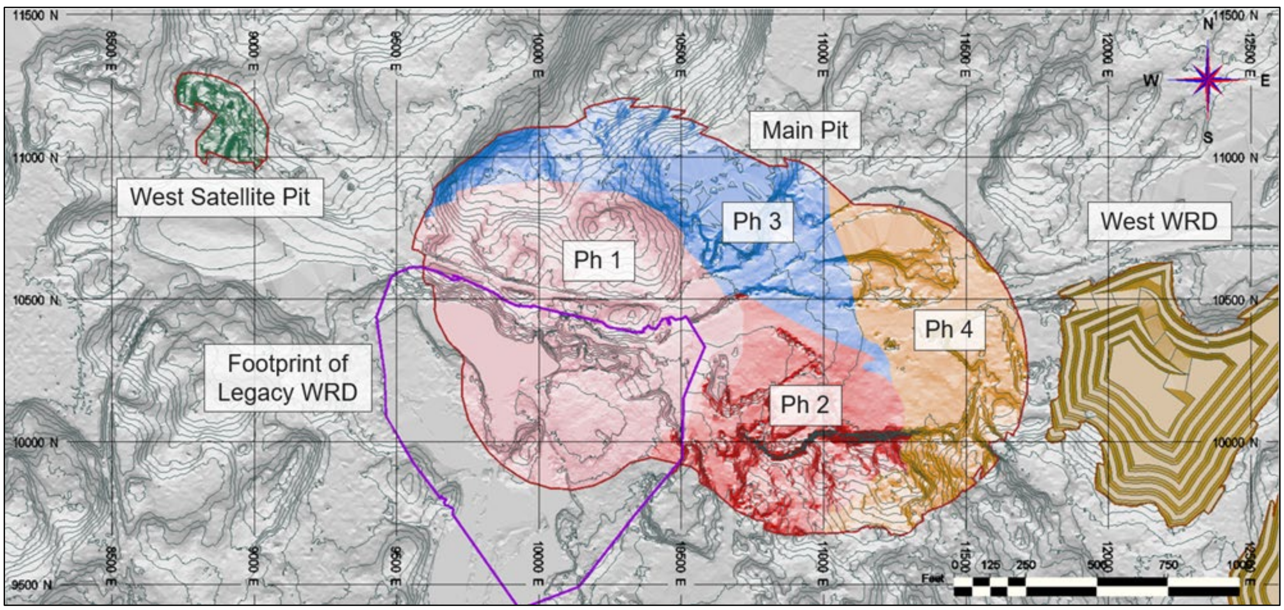
Storing waste in the Ellen pit would require approval from the pit's owner, Vale Canada Ltd., and may require additional permitting. The Ellen pit is an advantageous storage site because of its proximity to the Crean Hill open pit. Survey data of the Ellen pit was not available, so it is not known if the pit has sufficient capacity to store the required 19.7 Mt. If the pit does not have sufficient capacity, creating an additional waste storage site for excess waste will require additional permitting, site development costs, and possibly increase the cost of waste haulage.

Plans for water management, reclamation & closure, permitting, and geotechnical analyses have not been conducted at this point. These WRD designs are therefore conceptual in nature.

16.2.8 Mine Production Schedule

The Crean Hill open pit mine is composed of a main pit and the west satellite pit, as shown in Figure 16-10. The main pit is divided into four mining phases, and the west satellite pit is mined as one phase, as shown in Figure 16-10. Phase selection and design was guided by the results from the Whittle™ pit optimization process and the resulting Whittle™ pit-by-pit graphs.

Figure 16-10: Proposed Mining Phases for Crean Hill Open Pit Shown Without the Legacy WRD



Phase 1 was designed to be the starter pit because it contains material accessible at a relatively low-strip ratio. Phase 2 was designed to primarily be a waste stripping pit that can be mined while Phase 1 is developed. Phases 3 and 4 are much larger and require an advance pre-stripping effort to support sustained production.

Considering the size of the pit and limited working space, a nominal production rate of 3,000 tons per day was selected for the development of the mine production schedule.

Another important feature of the Crean Hill mining area is the existence of a large legacy WRD close to and partially covering the proposed Open Pit mining area. For planning purposes, the removal of this legacy WRD was considered in conjunction with the development of the open pit itself. See section 16.2.4 for more information regarding the legacy WRD.

16.2.8.1 SCHEDULE INPUTS AND DESIGN BASIS

It was assumed that all pre-requisite work, including permitting, detailed design, and procurement, required to mine the open pit would be completed in time to begin surface operations in November 2024.The pit was scheduled with a target ore production rate of 3,000 Tons per day (2,720 tonnes per day) at 360 operating days per year. The production schedule has been prepared on a quarterly basis from Q4 2024 through 2026, and annually thereafter.

The Crean Hill deposit has an extensive network of legacy mine workings with backfill, therefore some ore dilution and ore loss were anticipated. Stantec depleted ore from the block model where mined out stopes are known to exist. A detailed dilution and ore loss study has not been conducted at this time. Stantec has assumed an ore dilution rate of 5% and a mining recovery rate of 95% for this PEA.

As discussed in Section 16.2.2, the surface mine was designed to use relatively small mining equipment. One of the advantages to using smaller trucks and loading units is that a lengthy fleet assembly and commissioning period can be avoided because these units are relatively quick to assemble. Opting for a smaller fleet will allow for production to ramp-up quickly. It was assumed that a contractor will have the required equipment on-site and ready to support the production rates required in this schedule.

Stantec has planned for a maximum of three loading units, consisting of two excavators and one FEL. One loading unit is planned to be introduced to the operation per quarter, for the first three quarters. The loading unit requirement and associated loading rate estimates are presented in Table 16-10. The open pit schedule is discussed in Section 16.2.5.2.

Table 16-10: Loading unit requirement for initial production.

Schedule Period	Number of Excavators	Number of FEL(s)	Combined Loading Rate *(Mtpa)*
Q4 2024	1	0	5.22
Q1 2025	1	1	9.56
Q2 2025	2	1	14.79

16.2.8.2 PRODUCTION SCHEDULE DETAILS

Prior to commencing production from the main pit, the majority of the legacy WRDs must be relocated to the West and East Dump sites. This material is scheduled to be moved from Q4 2024–Q2 2025, which will allow ore production to begin in Q2 2025 from the Phase 1 pit. Material in the legacy waste rock pile has already been mined, so it will only need to be re-handled since blasting will not be required.

Initial ore production is focused on the Phase 1 pit from Q2–Q4 in 2025, with surplus production capacity focused on pre-stripping waste from phases 2–4. An intense pre-stripping effort will be required in 2025 to ensure steady ore production until the end of the Project because phases 2–4 have a higher strip ratio. The west satellite pit is relatively small compared to the main pit and will be entirely mined-out during Q2 2025.

Mining rates will peak in 2025 at 13.5 Mtpa. Material movement rates maintain a steady 3.6–3.7 Mt per quarter until declining to 1.9 Mt per quarter in Q4 2026 when most of the pit waste has been mined.

Despite declining mining rates, ore production rates will stay steady at 960–980 ktpa (kilo tonnes per annum) from 2026–2029.

Phase 2 is a high-strip pushback that is primarily focused on waste mining. Phase 2 mining will be completed in Q1 2026. Phase 1, the initial ore production pit, will be completely mined-out in Q3 2026, at which point the larger phase 3 pushback will become the primary ore source until it is completely mined-out in 2027.

Accessing the bottom of phase 3 requires haul trucks to travel through five consecutive switchbacks that will significantly increase the cycle times. Accessing the bottom of phase 3 via a typical spiral pit ramp is expected to substantially reduce ore recovery at the bottom of the pit but would likely reduce haulage costs. An access trade-off study should be considered during future work to investigate potential value savings by re-configuring the pit access plan.

Mining in phase 4 will begin in Q2 2026 and will be focused on waste stripping until Phase 1 is completed. Phase 4 will be accessed via two highwall ramps that meet at the 10,792 ft bench. Access to the bottom of the pit below 10,792 ft will be possible via one spiral ramp instead of the switch backs in phase 3. Phase 4 is scheduled to be completed in 2029. End-of-period plans have been prepared in Section 16.2.3.1 to show the detail of the surface mining sequence.

The open pit production schedule is summarized in Table 16-11 and Figure 16-11.

Figure 16-11: Crean Hill Open Pit Production Schedule Graphic

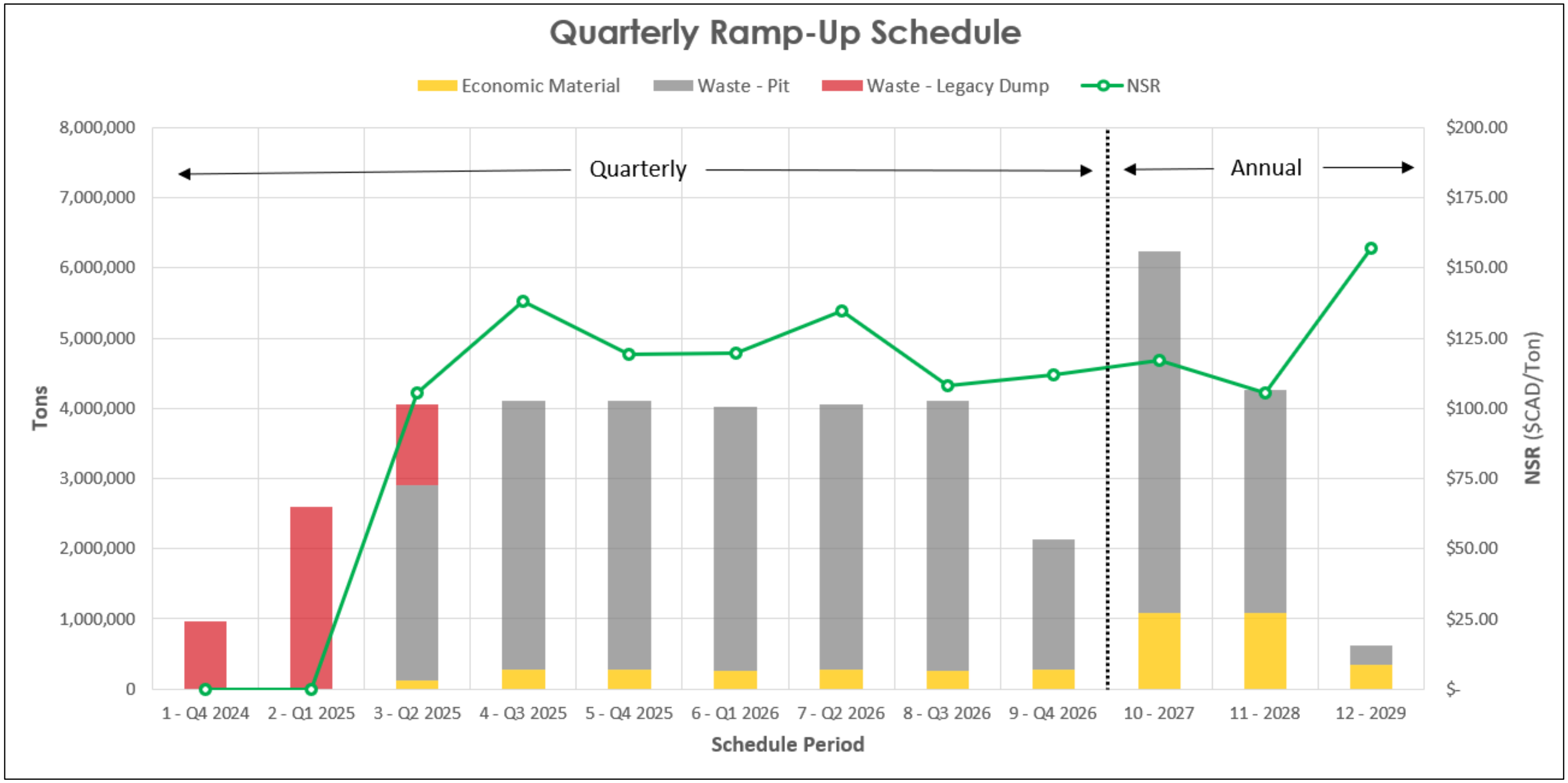


Table 16-11: Crean Hill Open Pit Production Schedule

Item	Units	Q4 2024	Q1 2025	Q2 2025	Q3 2025	Q4 2025	Q1 2026	Q2 2026	Q3 2026
Ore	Metric Tonnes	0	0	103,762	246,953	246,953	241,584	244,269	229,280
Legacy Waste	Metric Tonnes	871,768	2,357,689	1,047,924	0	0	0	0	0
Pit Waste	Metric Tonnes	0	0	2,534,968	3,480,214	3,480,214	3,404,557	3,442,386	3,497,887
Total Tonnage	Metric Tonnes	871,768	2,357,689	3,686,654	3,727,167	3,727,167	3,646,141	3,686,654	3,727,167
NSR Value	$CAD/Tonne	0.00	0.00	116.11	152.42	131.25	132.12	148.61	119.03
Nickel Grade	%	0.00	0.00	0.42	0.65	0.55	0.44	0.61	0.43
Copper Grade	%	0.00	0.00	0.48	0.44	0.45	0.42	0.46	0.35
Cobalt Grade	%	0.000	0.000	0.014	0.022	0.019	0.015	0.021	0.015
Platinum Grade	ppm	0.00	0.00	0.23	0.31	0.20	0.81	0.25	0.56
Palladium grade	ppm	0.00	0.00	0.26	0.20	0.13	0.60	0.14	0.47
Gold Grade	ppm	0.00	0.00	0.13	0.13	0.08	0.37	0.15	0.24

Item	Units	Q4 2026	2027	2028	2029	Total
Ore	Metric Tonnes	246,953	979,760	982,445	306,430	3,828,389
Legacy Waste	Metric Tonnes	0	0	0	0	4,277,381
Pit Waste	Metric Tonnes	1,692,719	4,675,395	2,885,896	255,234	29,349,469
Total Tonnage	Metric Tonnes	1,939,672	5,655,156	3,868,340	561,664	37,455,239
NSR Value	$CAD/Tonne	123.17	129.17	116.15	173.02	131.05
Nickel Grade	%	0.52	0.54	0.44	0.73	0.53
Copper Grade	%	0.35	0.38	0.40	0.48	0.41
Cobalt Grade	%	0.018	0.019	0.018	0.024	0.019
Platinum Grade	ppm	0.24	0.26	0.45	0.33	0.36
Palladium grade	ppm	0.19	0.14	0.24	0.16	0.22
Gold Grade	ppm	0.14	0.15	0.18	0.18	0.17

16.2.8.3 END-OF-PERIOD MAPS

Figure 16-12 through Figure 16-20 show the progression of Open Pit mining. The figures do not show the construction of surface infrastructure, waste rock dump development, or any other features external to the pit. Elevation contours of topography and the open pit are presented in 5 ft (1.5 m) and 16.4 ft (5 m) intervals respectively.

Figure 16-12: Original Topography With Pit Outline (Pre-Mining)

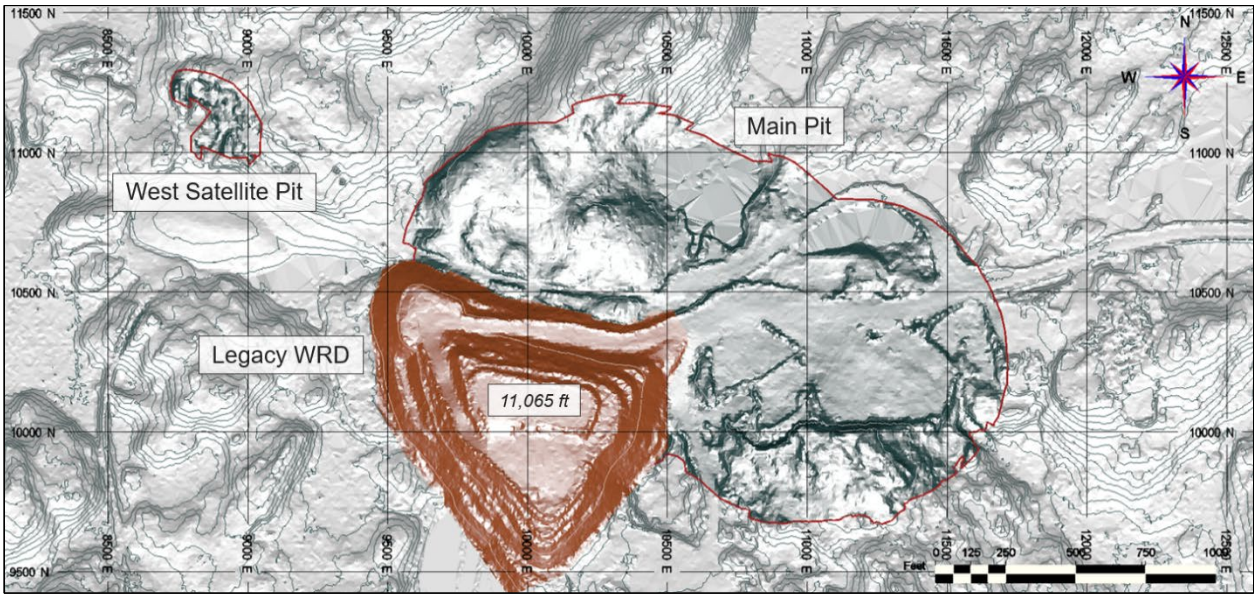


Figure 16-13: End of Q2 2025

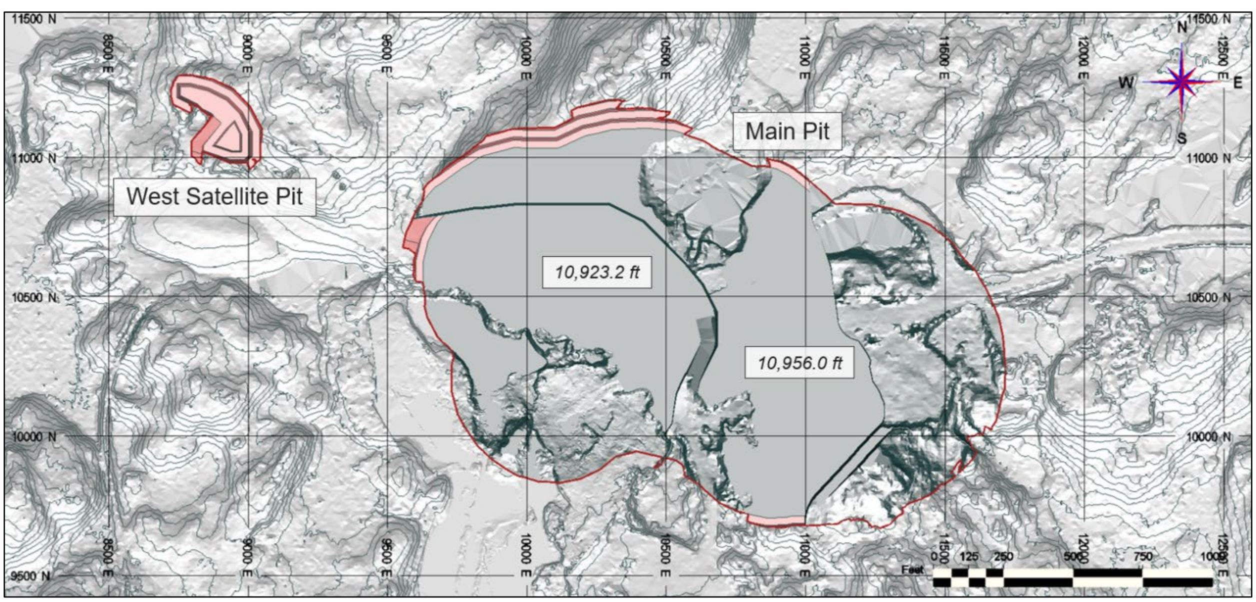


Figure 16-14: End of Q4-2025

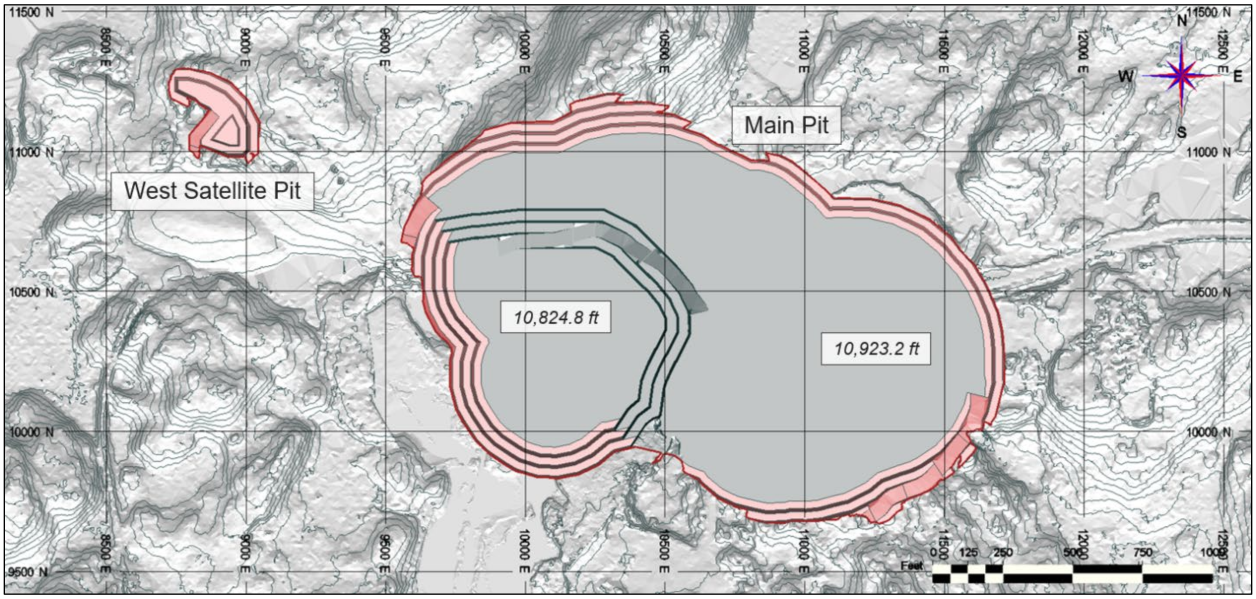


Figure 16-15: End of Q2-2026

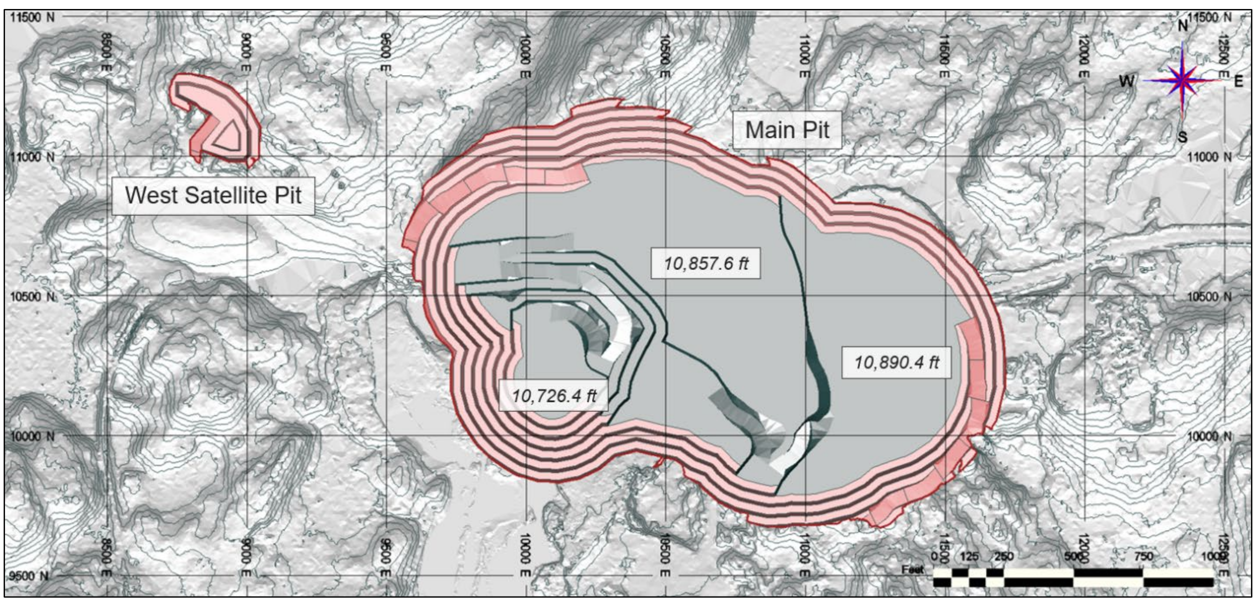


Figure 16-16: End of Q4-2026

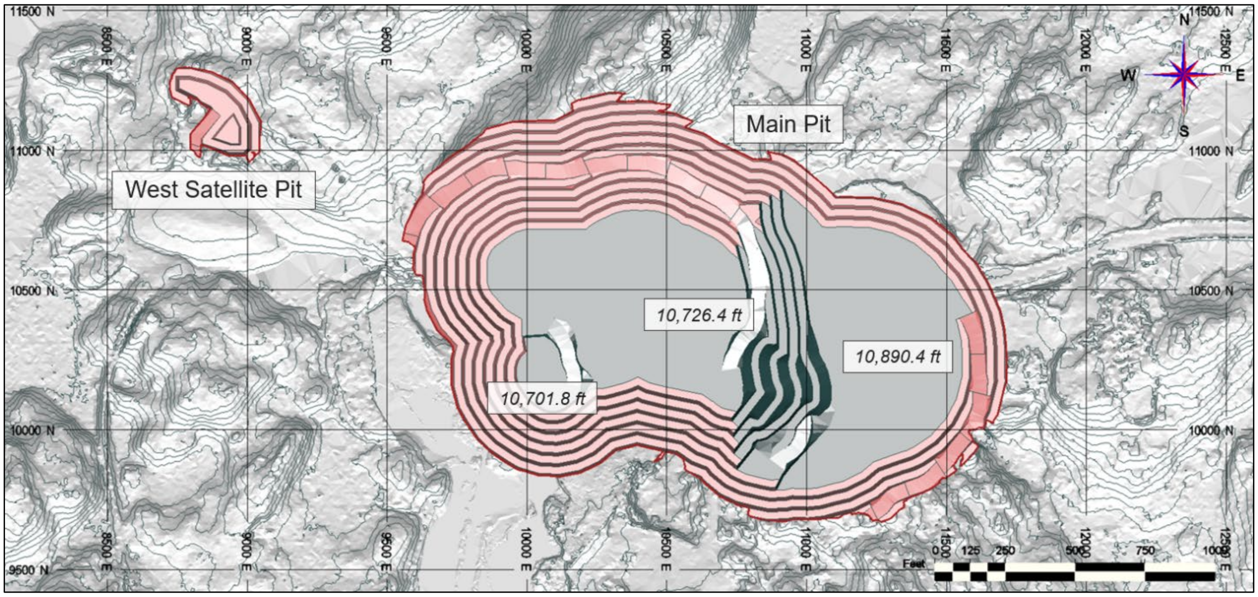


Figure 16-17: End of 2027

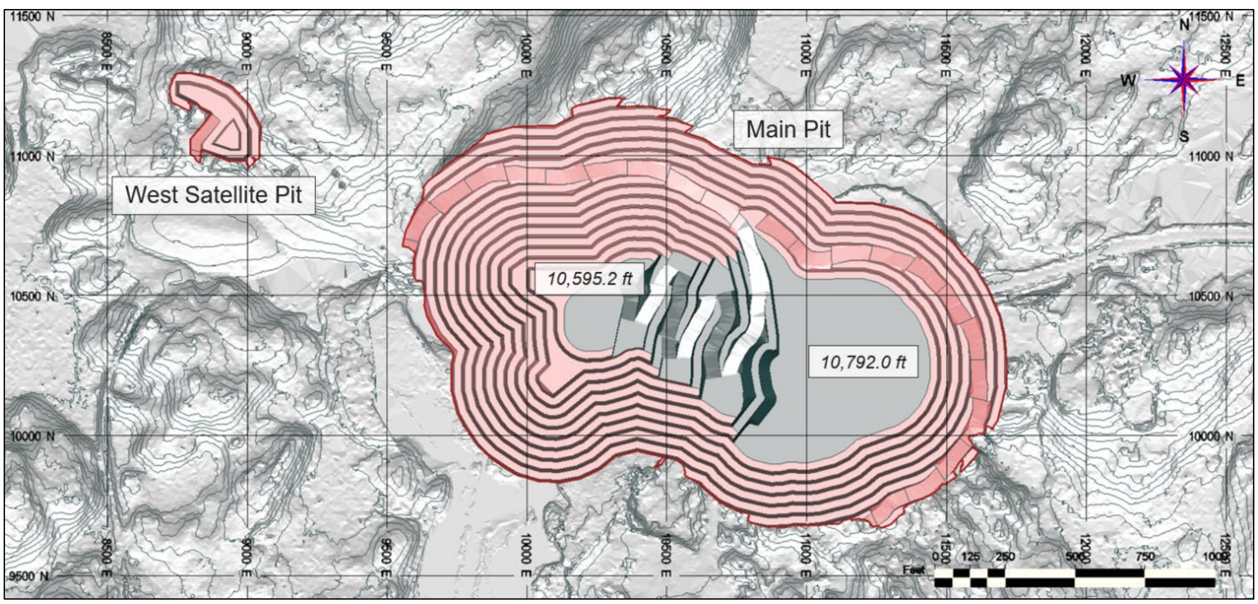


Figure 16-18: End of 2028

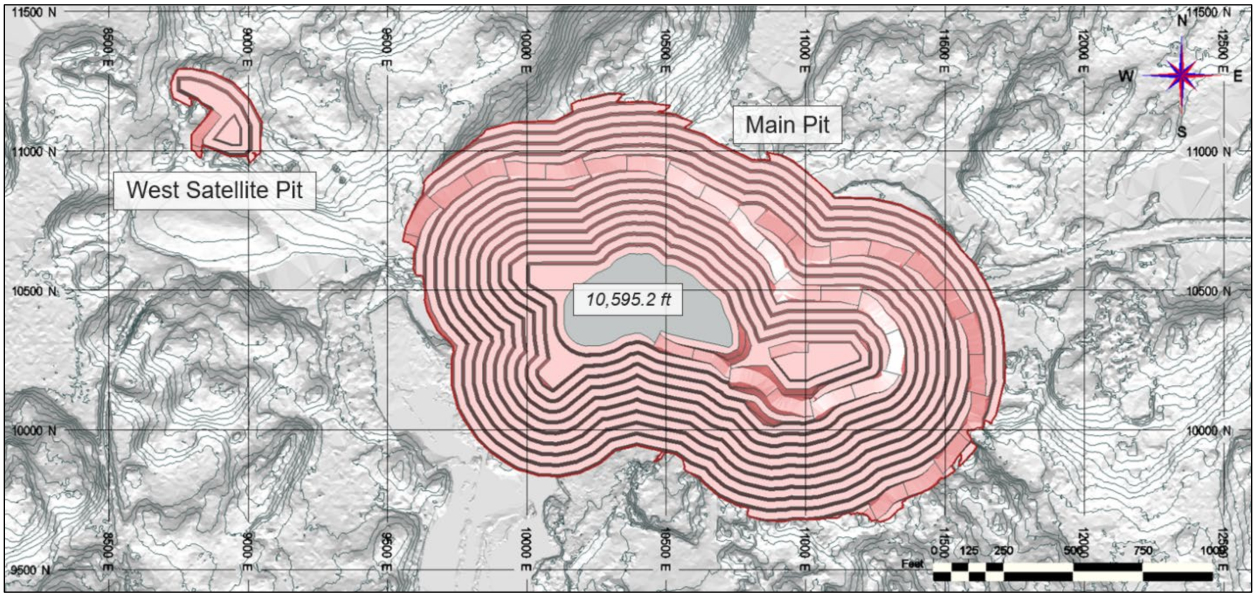


Figure 16-19: End of 2029

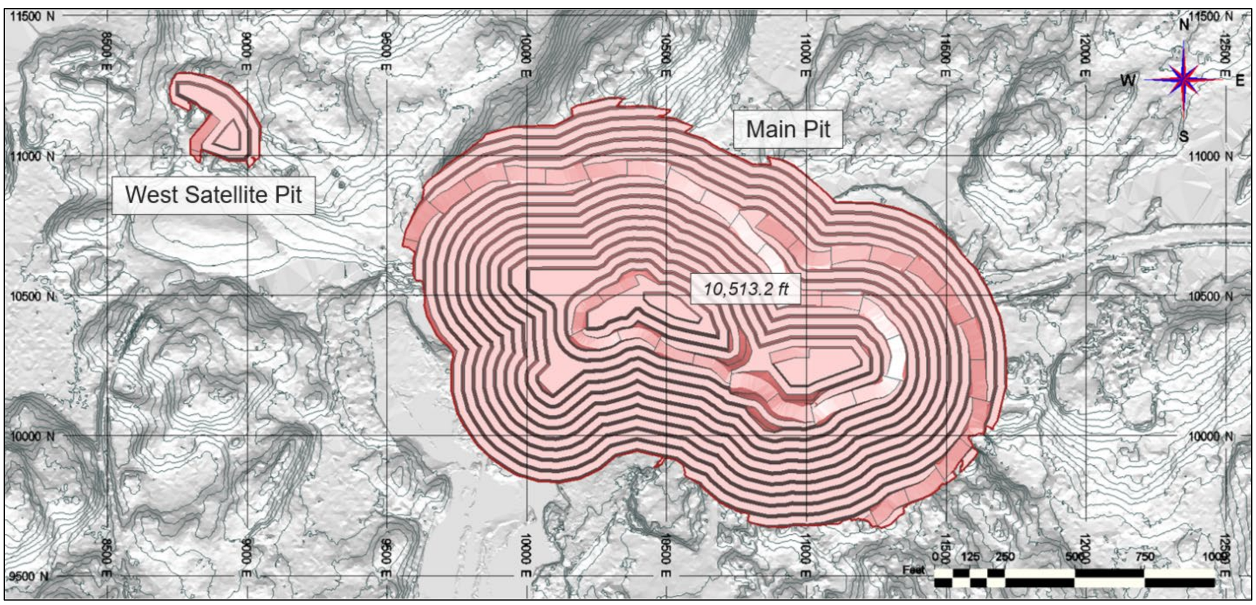
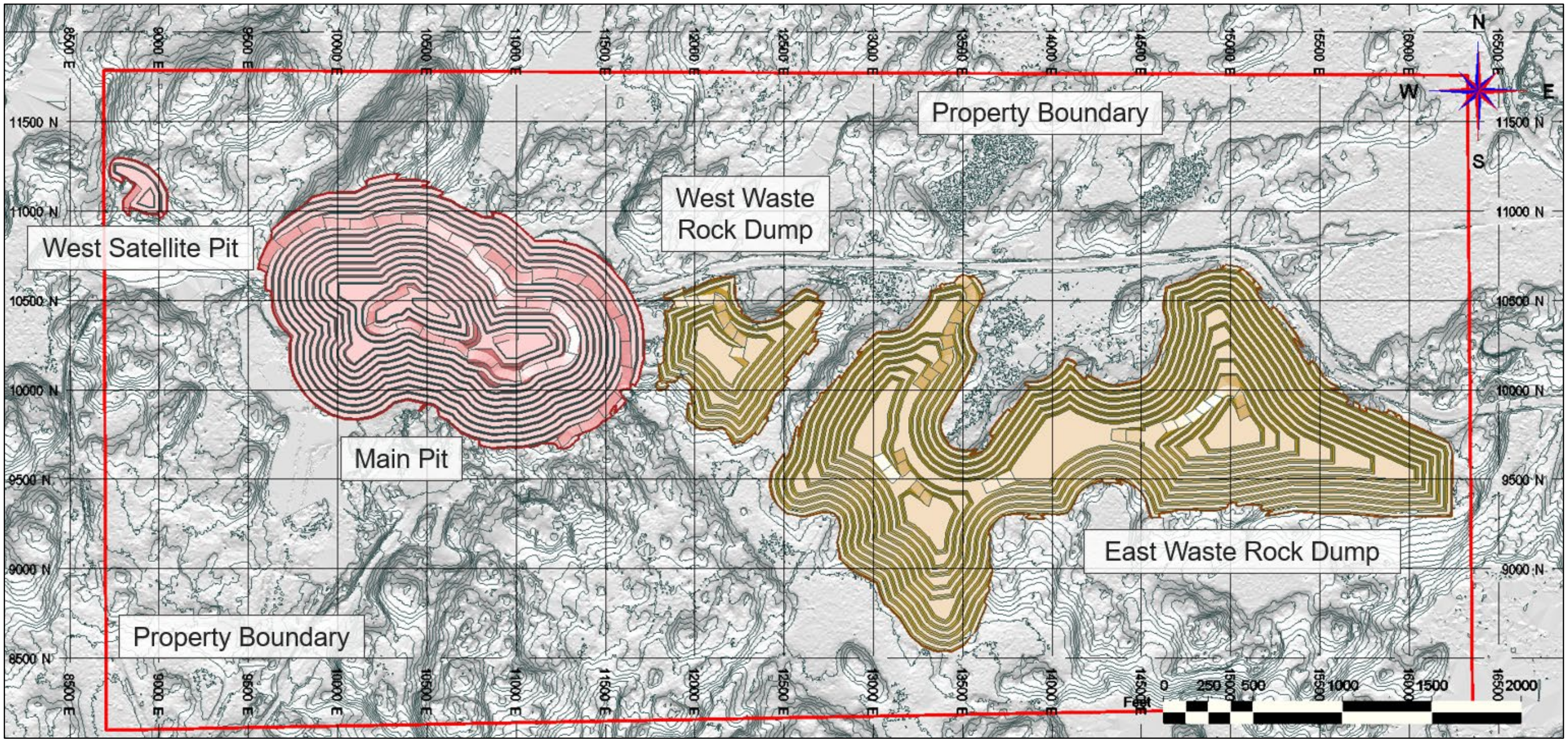


Figure 16-20: Crean Hill Waste Rock Dumps and End-of Open Pit Mining



16.2.9 Open Pit Mining Capital

Since the open pit life is limited to three years, a contract mining operation is planned. Therefore, capital for the open pit component of the Crean Hill mine is limited to dewatering equipment and a capital contribution to the crushing infrastructure.

16.3 UNDERGROUND MINING

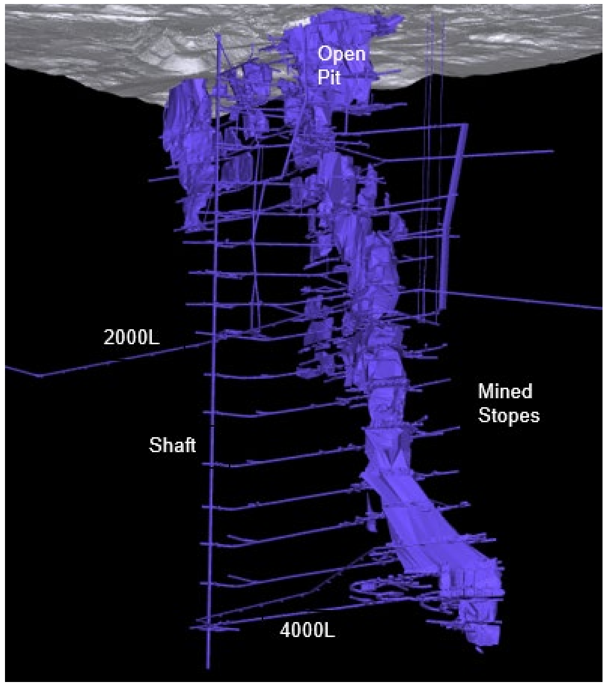
The existing Crean Hill underground mine workings have been flooded since operations ceased in approximately 2002. The stopes that were mined through to the surface or into open pits have been backfilled (from the surface) with unconsolidated waste rock. Shafts and raises to surface have been capped with concrete at the surface collars. The condition of the existing underground excavations is not known but are anticipated to be in reasonable condition.

The original shaft was approximately 2,000 ft deep to just below the 2000 Level (2000L) and was later deepened to just below the 4000L (approximately 4,180 ft below surface). The shaft is understood to have been equipped with five compartments (two skips, a cage, a counterweight, and services and a ladderway). The historic sublevels in the upper mine above 2000L were spaced at approximately 200-ft intervals and developed from the shaft as track drifts. There was no internal ramp system connecting between levels. In the upper levels the shrinkage stoping mining method was used, and stopes are assumed to be left open or partially backfilled with unconsolidated waste rock.

Below 2000L, the primary mining method was Vertical Retreat Mining (VRM) also known as Vertical Crater Retreat (VCR) with sublevel spacing ranging from approximately 250 ft to 300 ft and no internal ramp system. The VRM stopes were mined as primaries or secondaries with hydraulic sandfill used to fill the voids. The primary stopes were filled with consolidated sandfill and the secondary stopes were filled with unconsolidated sandfill. It is understood that some secondary stopes were mined in panels to limit the sandfill exposure of the primary stopes.

The 4000L area included a crusher and conveyor to the shaft loading pocket. The existing lateral development excavations outside the stopes are assumed to be accessible, however all previous underground infrastructure is assumed to be not available. An isometric view of the existing mine workings (development and stopes) is shown in Figure 16-21.

Figure 16-21: Existing UG Mine Workings (Isometric view)



Existing information and data for the Crean Hill Mine are available in Imperial Units of measure. Imperial units were used during the PEA mine design process and converted to metric for this report where appropriate.

The original names of sublevels when the mine was in operation are understood to be based on approximate depth below surface. For the PEA mine design, the levels have been named based on approximate depth in feet below surface. For example, 2000L is approximately 2000 ft below surface. The elevation at surface is 10,960 RL (Reduced Level).

The resource model along with the corresponding wireframe was prepared by SGS Geological Services and includes indicated and inferred resource classifications. A Net Smelter Return (NSR) field was added to the resource model and used to identify the target areas for mine planning.

The mineralized resource densities are included in the resource model. For calculations where data is not directly taken from the model, average in-situ densities will be used and are summarized in Table 16-12.

Table 16-12: Mineralized Resource and Waste Rock In-Situ Densities

Item	In-Situ Density Short tons/cubic foot	In-Situ Density Metric tonnes/cubic metre
Mineralized Resource	0.094	3.01
Waste Rock	0.088	2.82

16.3.1 Geomechanical

Geomechanical data from previous reports and studies was used to establish rock mass properties for analysis of stope stability and for the stability of pillars against historic mined out stopes that may or may not contain backfill.

16.3.1.1 LONGHOLE STOPE STABILITY

For the stope sizing and stope stability assessment, the Modified Stability Graph method was used. The Modified Stability Number (N') was estimated using the information from previous reports and plotted against the stope surface hydraulic radius (area/perimeter ratio). For longhole stopes that are approximately 82 ft (25 m) high, 66 ft (20 m) wide along strike, and 50 ft long (15 m) (from hangingwall to footwall) the median N' value falls within the stable zone for all stope surfaces (with no additional support) except if the stope surface is in a dyke structure. The Modified Stability Graph plots for the stope sidewall, back, and hangingwall/footwall are shown in Figure 16-22, Figure 16-23, and Figure 16-24.

Figure 16-22: Stope Stability – Stope Sidewall

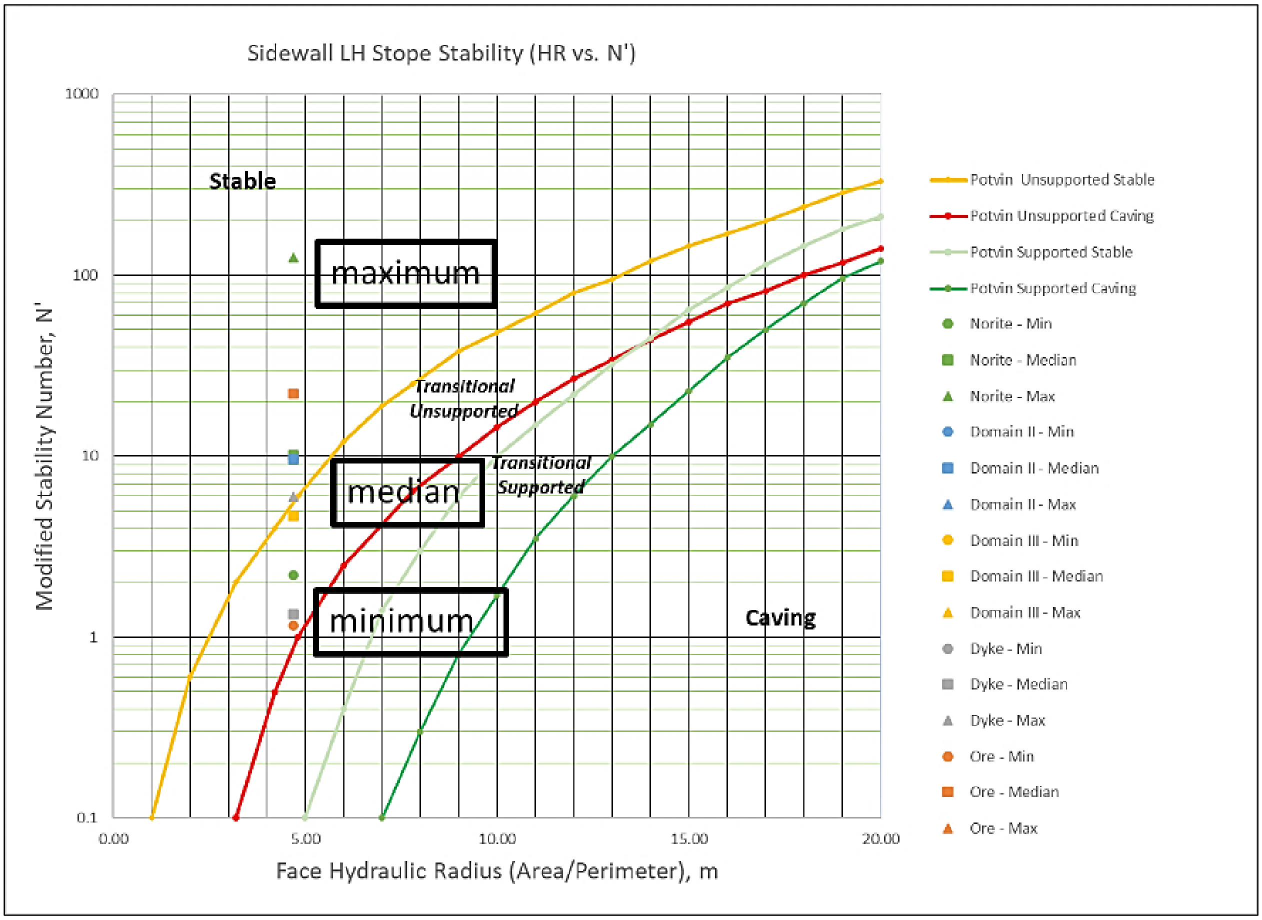


Figure 16-23: Stope Stability – Stope Back

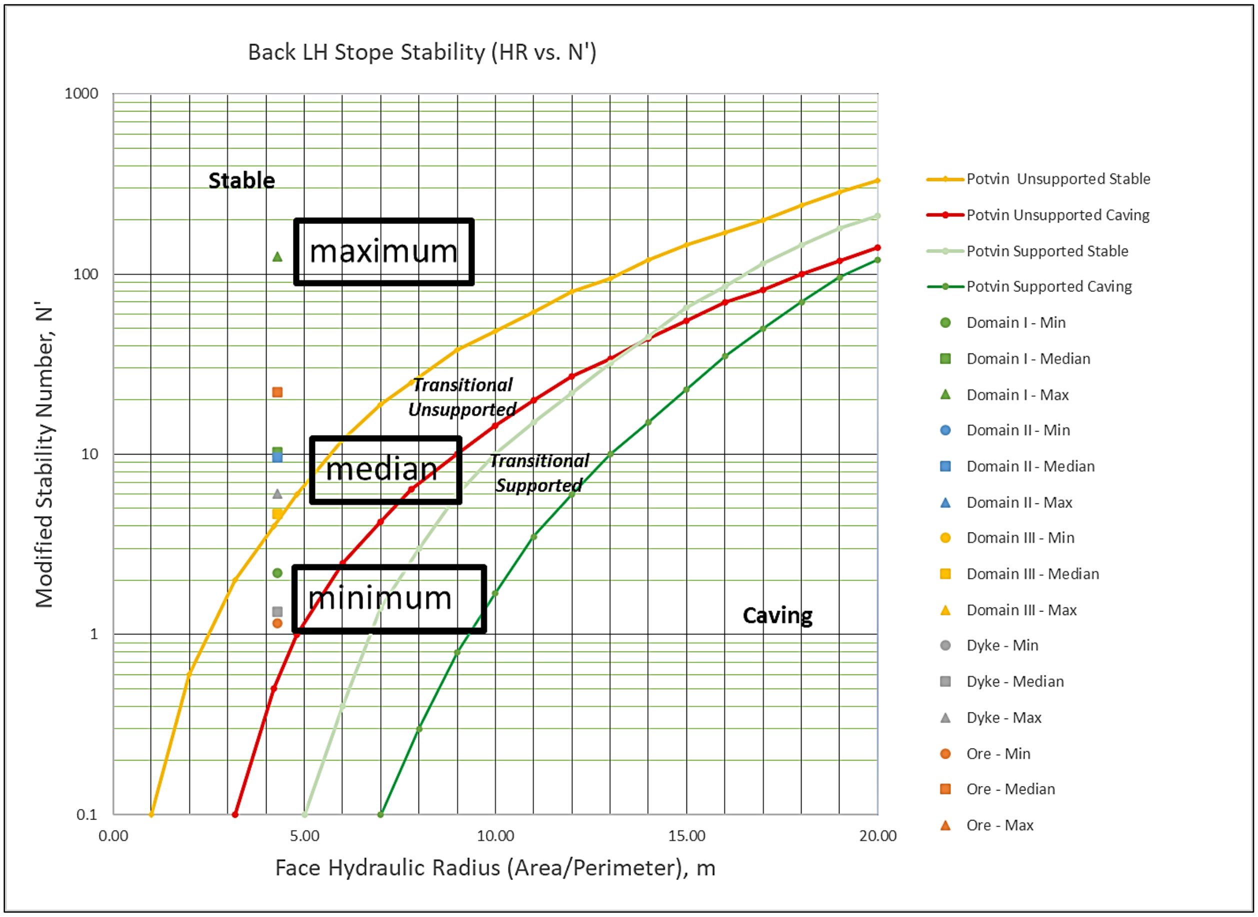
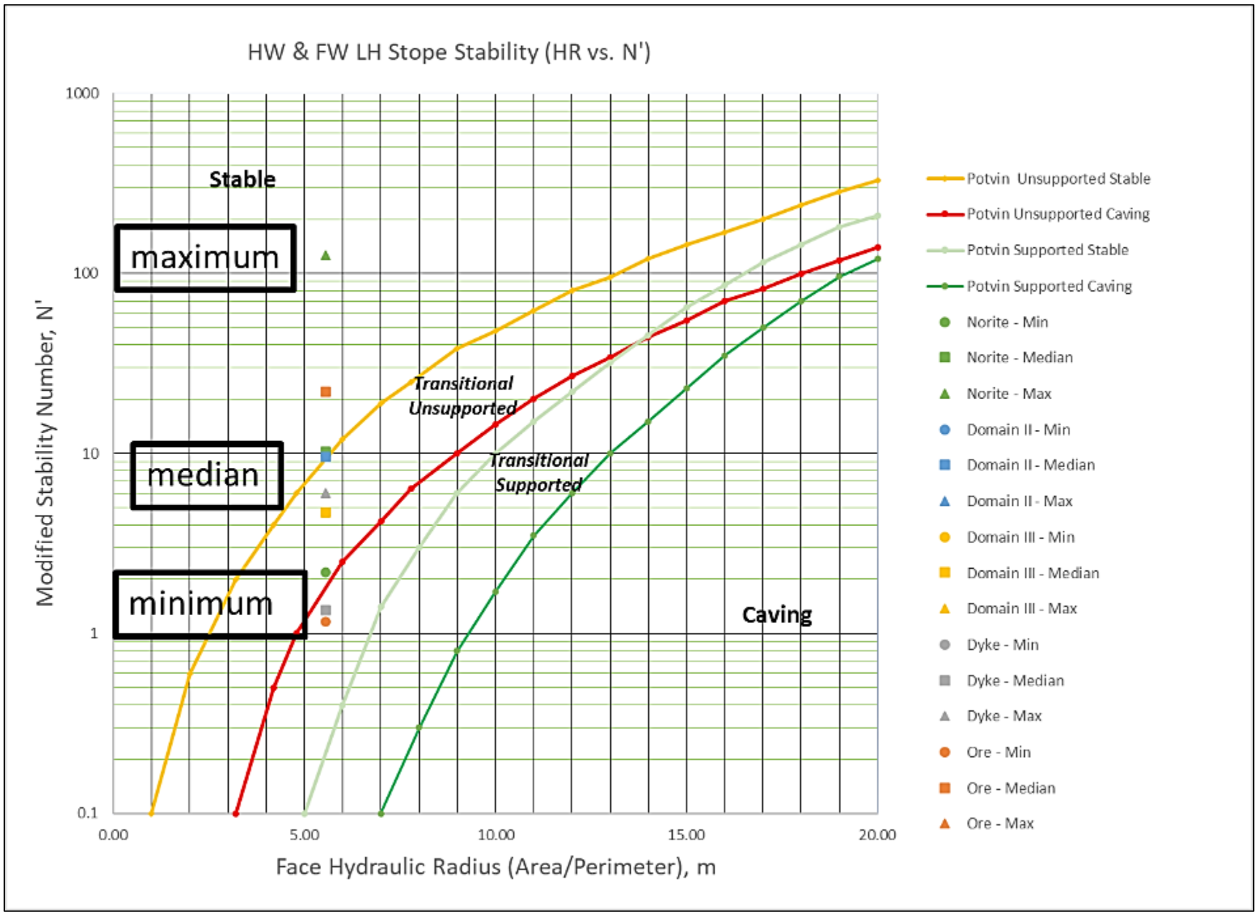


Figure 16-24: Stope Stability – Stope Hangingwall/Footwall



16.3.1.2 PILLAR DESIGN

There is mineralized resource adjacent to the historic mined out stopes. The backfill status of the stopes is not known above 2000L, and stopes could be open, partially backfilled with unconsolidated fill, or fully backfilled with unconsolidated fill. Below 2000L, the stopes are known to have been backfilled with sandfill. The stopes below 2000L were mined in a primary and secondary sequence (with the primaries containing sufficient binder to mine against), however it is not known which stopes were primaries and filled with consolidated sandfill.

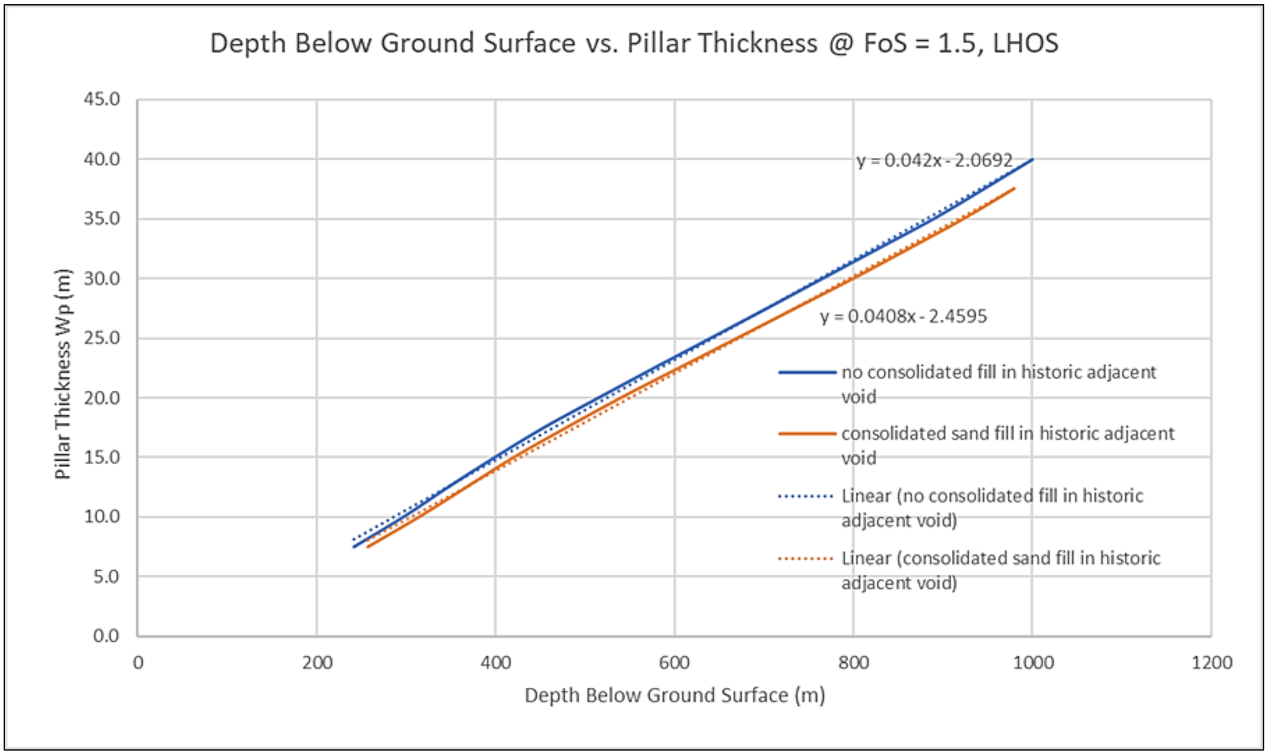
The size of pillar to leave between a new stope and the historic stopes was assessed using modelling in Rocscience RS3 software. Four cases were assessed with modelling:

- Case 1 – Mine a longhole stope adjacent to a historic stope that is empty or filled with unconsolidated backfill.
- Case 2 – Mine a cut and fill stope adjacent to a historic stope that is empty or filled with unconsolidated backfill.
- Case 3 – Mine a longhole stope adjacent to a historic stope that is filled with consolidated sandfill.

- Case 4 – Mine a cut and fill stope adjacent to a historic stope that is filled with consolidated sandfill.

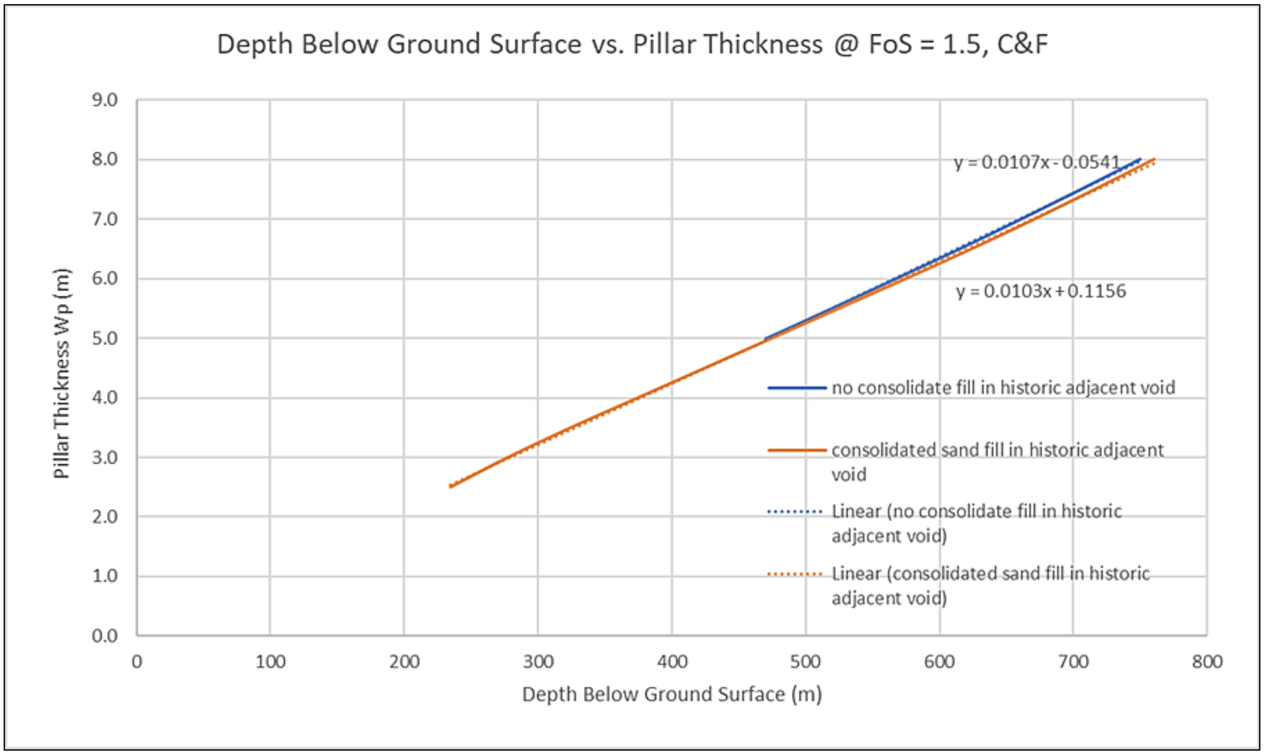
The pillar thickness was estimated at a factor of safety (FoS) of 2.0, 1.5, and 1.0 using the method established by Lunder et.al., 1997. This method accounts for the effects of field stress on rock mass strength. In general, rib pillar dimensions increase as stress increases at depth. The pillar thickness required for longhole stopes near historic mined out areas at a factor of safety of 1.5 is demonstrated in Figure 16-25.

Figure 16-25: Longhole Stope Case 1 and Case 3 Pillar Thickness (FoS – 1.5)



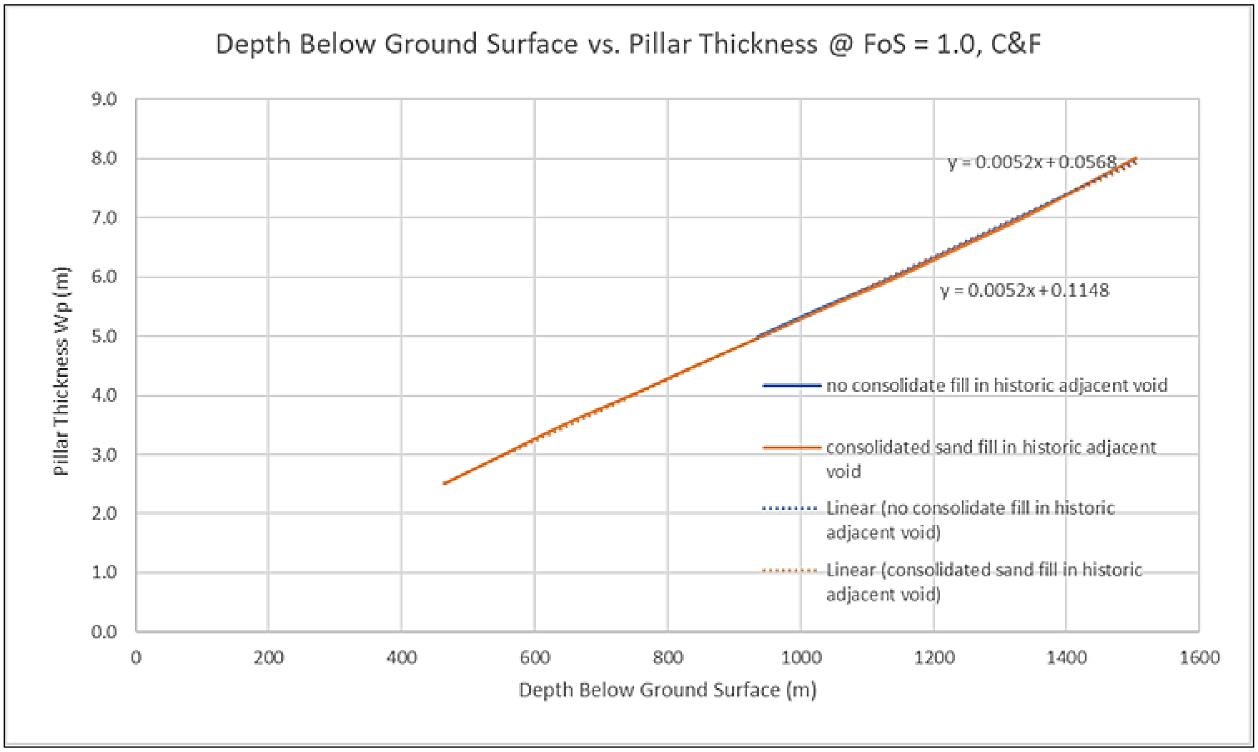
The pillar thickness required for cut and fill stopes near historic mined out areas at a factor of safety of 1.5 is demonstrated in Figure 16-26.

Figure 16-26: Cut and Fill Case 2 and Case 4 Pillar Thickness (FoS = 1.5)



The pillar thickness required for cut and fill stopes near historic mined out areas at a factor of safety of 1.0 is demonstrated in Figure 16-27.

Figure 16-27: Cut and Fill Case 2 and Case 4 Pillar Thickness (FoS = 1.0)



The resource adjacent to these historic stopes is generally of higher grade and value. To maximize recovery of the high-value resource, selective cut and fill methods will be used to allow mining closer to the historic workings. It is anticipated that systematic probe hole drilling and ground monitoring will be implemented to maximize resource recovery. When mining cut and fill near historic stopes a 16.4-ft (5.0 m) pillar has been left in place. If reliable backfill records for consolidated backfill placement in historic mining areas are available, there is potential to mine against the fill.

There are methods for guidance (such as Mitchell's Method) on how much historic fill can be safely exposed by new mining. Also, historic backfill may be sampled by core drilling and core tested for compressive strength. For the PEA, it has been assumed that some of the 16.4-ft (5.0 m) pillars below 2000L will be recovered.

16.3.1.3 SHAFT STAND-OFF DISTANCE FROM OPEN PIT CREST

The stand-off distance from the proposed pit crest to the existing shaft location considers the crown pillar thickness and the influence angle of subsidence. Ground disturbance due to subsidence may be initiated below the crown pillar and critical mine infrastructure will not be placed within this horizontal extent of subsidence influence. The extent of this influence is a geometric relationship where the angle of influence projected from underground workings nearest the pit bottom daylights when projected to pit crest elevation, in relation to the pit crest. The existing shaft location relative to the pit is not within this zone of influence.

16.3.1.4 GROUND SUPPORT

For ground support, a standard pattern of 8-foot-long resin rebar installed on a 4-ft by 4-ft pattern with welded-wire mesh screen has been assumed for all development. To account for larger spans at intersections and areas of poorer quality ground an allowance for applying shotcrete to 10% of lateral development has been assumed.

16.3.2 Preliminary Cut-Off Value

The preliminary cut-off Value for the underground mine design was estimated based on the cost assumptions summarized in Table 16-13. The preliminary cut-off values assume third party milling costs.

Table 16-13: Preliminary Cut-Off Value ($/short ton) for Underground Mine Planning

Item	$/ton Bulk Mining Target COV	$/ton Bulk Mining Incremental COV	$/ton Selective Mining Target COV	$/ton Selective Mining Incremental COV
Operating Development	$5.00	$4.00	$6.00	$6.00
Stoping	$8.00	$8.00	$18.00	$18.00
Backfill	$9.00	$9.00	$9.00	$9.00
UG Ore Handling to Surface Pad	$4.00	$4.00	$4.00	$4.00
UG Waste Rock Handling	$2.00	$0.50	$2.00	$1.00
Waste Rock Handling on Surface	$1.00	$0.00	$1.00	$0.00
Power	$6.00	$4.00	$6.00	$4.00
Mine Air Heating	$2.00	$2.00	$2.00	$2.00
Fixed Plant Maintenance	$6.00	$4.00	$6.00	$4.00
Definition Diamond Drilling	$2.00	$2.00	$2.00	$2.00
Indirect Mobile Equipment Operating	$5.00	$4.00	$5.00	$4.00
Indirect UG Mine Labour (surface /UG)	$21.00	$16.12	$21.00	$16.12
Sustaining Capital	$5.00	$0.00	$5.00	$0.00
Administrative Labour	$2.28	$0.00	$2.28	$0.00
Surface Building Maintenance	$1.00	$0.00	$1.00	$0.00
Services	$0.25	$0.00	$0.25	$0.00
Surface Support Equipment	$0.25	$0.00	$0.25	$0.00
Misc.	$1.00	$0.00	$1.00	$0.00
Surface Handling/Crushing	$2.00	$2.00	$2.00	$2.00
Haulage to Mill	$5.00	$5.00	$5.00	$5.00
Milling/Smelting	$35.38	$35.38	$35.38	$35.38
Total UG Mine Operating Cost	**$123.16**	**$100.00**	**$134.16**	**$112.50**

The mine design will be largely based around mining stopes with an NSR above the target cut-off values with incremental material in the proximity also included in the mine plan.

16.3.3 Mine Access

Access to the underground workings will be established in a phased approach. Initially, a boxcut and portal will be constructed, and an Access Ramp developed to support an Advanced Exploration (ADEX) program that includes an underground bulk sample and diamond drilling. Following the ADEX program, the Access Ramp will be extended to support initial production from the upper portion of the mine (above 2000L) while the existing shaft is being reconditioned and re-equipped for production and servicing the mine below 2000L down to 4000L. The Access Ramp and internal ramp system will extend to the bottom level of the mine at 4350L.

16.3.3.1 BOXCUT AND PORTAL

The boxcut and portal will be established at an outcrop that has a highwall face that was previously excavated and supported for other purposes. The proposed boxcut locations will be investigated for hazards (such as historic raise or borehole) that may exist but are not shown on existing mine models. The planned portal location is shown in a satellite view of the site in Figure 16-28.

Figure 16-28: Planned Portal Location (not to scale)

A short access road will be established to the portal location and an area prepared to make a pad for infrastructure and services and temporary waste rock and mineralized resource stockpiles. The road leading into the portal will have a gradient and ditching to minimize the catchment of surface water run-off into the ramp. There will be a small sump established a short distance down the ramp to collect any water that may enter the ramp. The portal boxcut design assumptions are summarized in Table 16-14.

Table 16-14: Portal Boxcut Design Assumptions

Item	Assumption
Overburden / Fractured Rock	Outcrop / Bedrock
Competent Rock above Portal Face	1.0 x the Portal Socket Width
Boxcut / Portal floor gradient	Transition from +2% to -17%
Ramp Socket Dimensions	Ramp width + 3.0 feet (1.5 feet each wall) Ramp height + 1.5 feet
Ground Support	Patterned Resin Rebar and Chain Link Screen. Straps at the brow.
Portal Cover	None

16.3.3.2 ACCESS RAMP AND INTERNAL RAMPS

The upper portion of the Access Ramp will be developed with sufficient back height to accommodate two lines of 56-in diameter ventilation ducting and the largest piece of mobile equipment (while the duct is in place) anticipated to be a loaded 40T class haul truck. Once the ventilation system is established the ventilation ducting size will be smaller and the ramp back height can be reduced. The design assumptions for the Access Ramp and Internal Ramps are summarized in Table 16-15.

Table 16-15: Ramp Design Assumptions

Item	Access Ramp	Internal Ramps
Width Height	16.4 ft (5.0 m) 18.0 ft (5.5 m)	16.4 ft (5.0 m) 16.4 ft (5.0 m)
Planned Maximum Gradient Design Gradient for PEA (accounts for transitions at sublevels)	17% 15%	17% 15%
Safety Bays and Remuck Bays	Accounted for in a design allowance factor	

16.3.3.3 SHAFT

The existing shaft has a concrete cap and is flooded. The shaft extends to just below the 4000L (approximately 4,180 feet below surface) and is understood to have been previously equipped with five compartments (two skips, a cage, a counterweight, and services and a ladderway). The purpose and use of the existing shaft will evolve as the Project and mine life progresses. The shaft use at various stages of mine development is summarized in Table 16-16.

Table 16-16: Shaft Use at Various Stages of Mine Development

Shaft Use	Notes
Dewater flooded UG workings	Lower a pump down the shaft from surface.
Ventilation for initial Access Ramp Development	Access Ramp will connect to existing near-shaft development at various elevations to establish a ventilation loop.
Egress – 2000L to Surface Ventilation for Production above 2000L	Use an Alimak to inspect, repair, and establish egress for ramp production above 2000L. Provide ventilation for ramp production above 2000L
Production Shaft – Shaft bottom to Surface Egress – 4000L to Surface Ventilation	Recondition and re-equip the shaft to produce from below 2000L (shaft stations, loading pocket, etc.). Main Access/Egress from the mine. Ventilation.

The final Production Shaft will be used for skipping and movement of personnel and materials and provide key services. The assumptions for the Production Shaft facility are summarized in Table 16-17.

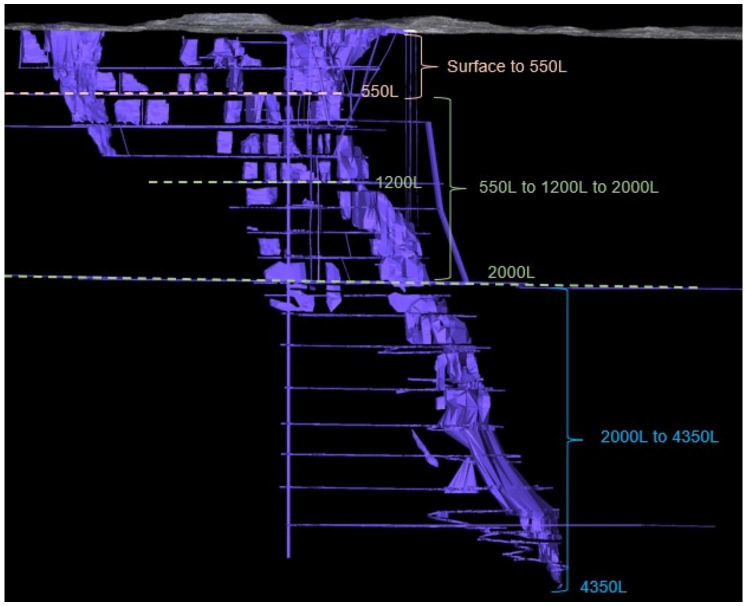
Table 16-17: Production Shaft Facility Assumptions

Item	Notes
Headframe	Steel structure Collar house Bins/Truck Loadout Chute to outside pad
Production Hoist Cage Hoist Hoisting capacity	15 ft diameter, Double Drum, 3,500 hp 15 ft diameter, Double Drum, 3,500 hp 4,500 tons per day from 4100 L
Configuration	2-Skip Compartments, 15-ton skips 1-Cage compartment 1-Cage Counterweight 1-Ladderway/Services
Services	Power & Communications cables Compressed Air, Process Water, Dewatering pipe
Shaft Stations	2000L, 2500L, 2850L, 3400L, 4000L
Loading Pocket	Approximately 4100L
Material Sizing	Rock breaker and Grizzly

16.3.3.4 DEWATERING EXISTING FLOODED WORKINGS

The existing underground mine voids (including the open pits that are connected to underground workings) are either backfilled with waste rock or sandfill or flooded with water. The mine will require dewatering of flooded workings ahead of advancing development. The volume to be dewatered was estimated from the mine model of existing voids and assumptions for the volume that may be displaced by backfill. The mine was divided into four dewatering elevations as shown in Figure 16-29.

Figure 16-29: Underground Dewatering Elevations



The assumptions of volume of voids backfilled and estimated dewatering volumes between each elevation are summarized in Table 16-18.

Table 16-18: Estimated Flooded Workings Dewatering Volume

Elevation Interval	Assumption	Estimated Dewatering Volume US gallons
Surface to 550L	Open Pits are backfilled from surface and any underground stopes connected to surface (and pits) are backfilled. 80% of Stope voids filled with waste rock. 0% of development voids filled with waste rock.	112.0 million
550L to 1200L	Shrinkage stopes did not require backfill as part of the mining method. 25% of Stope voids filled with waste rock. 0% of development voids filled with waste rock.	286.3 million
1200L to 2000L	Same as 550L to 1200L.	202.0 million
2000L to 4350L	VCR Stopes required backfill as part of the mining method. 90% of Stope voids filled with sandfill. 0% of development voids filled with waste rock.	171.1 million

Three dewatering approach options were considered, including the following.

- Dewater from Surface with a pump lowered in the shaft.
- Dewater from Surface with a pump lowered in a borehole drilled to intercept an underground void.
- Enter the shaft with an Alimak and repair the shaft and dewater simultaneously.

Lowering a pump in the shaft was the preferred approach selected for the PEA. The main considerations include the following.

- The borehole option will be difficult to drill to provide a vertical hole due to both surface and underground constraints and in finding a suitable location.
- The water vertical recede rate (at a reasonable pumping rate) will constrain and increase the cost of the Alimak option. It was determined the dewatering should be de-coupled from the shaft reconditioning.
- If the pump lowered in the shaft encounters an obstruction, dewatering to the obstruction can proceed followed by accessing the shaft with an Alimak to remove the obstruction.

The pumping assumptions for dewatering the flooded workings are summarized in Table 16-19.

Table 16-19: Dewatering Pumping Assumptions

Item	Assumption
Pump Rate	500 USgpm
Average Inflow Recharge Rate	50 USgpm
Effective Dewatering Rate	450 USgpm
Pumping Hours per Day	24 hours per day
Daily Volume Pumped	648,000 US gallons

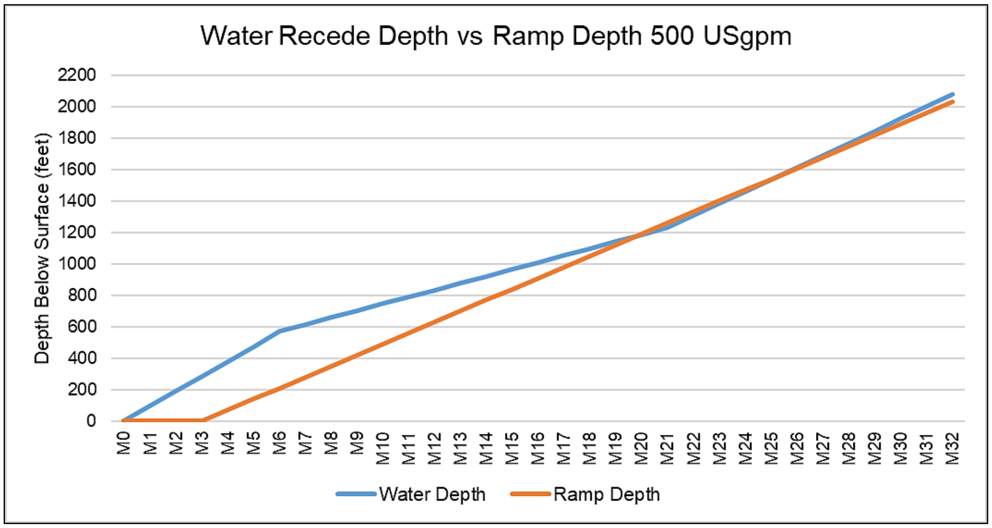
Based on the volume and pumping assumptions the following water average recede rates summarized in Table 16-20 were estimated.

Table 16-20: Estimated Water Level Recede Rate

Elevation Interval	Volume (US gallons)	Average Recede Rate Vertical feet/day
Surface to 550L	112.0 million	3.2
550L to 1200L	286.3 million	1.5
1200L to 2000L	202.0 million	2.6
2000L to 4350L	171.1 million	8.9

The water recede rate was compared to the estimated rate of Access Ramp vertical advance. Assuming the ramp will descend at a rate of 70 vertical ft per month and the dewatering program starts 3 months before ramp development, the water level will stay ahead of ramp development to 1200L. Below 1200L the water recede rate and ramp descend rate will be similar. The water recede depth versus the ramp advance depth are shown graphically in Figure 16-30.

Figure 16-30: Water Recede Depth versus Ramp Advance Depth



16.3.4 Development Methods

All ramp and lateral excavations will be developed using conventional drill and blast methods and diesel-powered mobile equipment. For larger excavations such as the maintenance shop, initial pilot drifts will be developed and a combination of wall slashing, floor benching, and back-slashing will be used to achieve the final dimensions. The development method assumptions are summarized in Table 16-21.

Table 16-21: Development Method Assumptions

Development Item	Assumption
Drill	2-Boom Jumbo Round Break 13.2 ft (4.0 m)
Blast	Anfo with packaged explosives for perimeter control and lifters Non-electric detonators Blast-at-will until production activities start
Muck	14T class Load Haul Dumper (LHD) Average tram distance approximately 250 ft (75 m) to a remuck
Ground Support	Mechanical Bolter 8-foot Resin Rebar 4 ft x 4 ft pattern Welded-wire mesh screen Allowance for Shotcrete – 10% of lateral development Face bolting
Haul	40T class underground haul truck Loaded with LHD at remuck entrance

16.3.4.1 LATERAL DEVELOPMENT

Sublevel development will be in the footwall rock. The typical infrastructure on a sublevel will include the following.

- Level access from the internal ramp system
- Remuck and truck loading area at the level entrance
- Water collection sump
- Electrical cut-out
- Stope accesses
- Ventilation raise access

On some sublevels, additional infrastructure will include the following.

- Refuge Stations
- Explosives and Detonators Storage
- Material Storage Bays
- Pump Stations
- Fuel Bays
- Maintenance Shop and Wash Bay
- Backfill slurry cut-out

The lateral development design assumptions are summarized in Table 16-22.

Table 16-22: Lateral Development Design Assumptions

Item	Level Access Waste Rock	Other Dev Waste Rock	Stope Crosscut and Sills
Width Height	16.4 ft (5.0 m) 16.4 ft (5.0 m)	14.7 ft (4.5 m) 14.7 ft (4.5m)	14.7 ft (4.5 m) 14.7 ft (4.5 m)
Planned Gradient (for drainage) Design Gradient for PEA	+/- 2% 0%		
Stand-Off Distance from Ramps to Mineralized Zone	Approximately 100 ft (30.0 m)		

Existing development on sublevels above 2000L were developed as track drifts at approximately 10.0 ft to 12.0 ft wide x 11.0 ft to 12.0 ft high. At lower levels of the mine, existing sublevel development is larger, up to 14 ft wide x 13.0 ft high. Use of these existing excavations will require slashing to the required dimensions and installation of new ground support.

16.3.4.2 VERTICAL DEVELOPMENT

Raises will be required for ventilation, egress, and muck passes. The assumptions and design criteria for raises are summarized in Table 16-23.

Table 16-23: Raise Development Design Assumptions and Criteria

Item	Raise < 100 ft^2 Area	Raise > 100 ft^2 Area
Excavation Method	Alimak*	Raisebore
Supported	Yes (due to Alimak)	Yes, if below 2,000 ft from surface Yes, if greater than 15-ft diametre Yes, if equipped with Escapeway

* Raises less than 100 feet long and not supported can be drop raised.

16.3.4.3 DEVELOPMENT OVERBREAK AND DESIGN ALLOWANCE

Overbreak and design allowance factors have been applied to the neat quantities for development in waste rock. Overbreak accounts for the excavation face area breaking beyond the planned dimensions. Design allowance accounts for miscellaneous excavations that are not shown in the Deswik model (such as remuck bays, safety bays, storage bays, electrical cut-outs, slashes at intersections, back slashes, etc.). These factors will be applied in the Deswik model and reported in the overall quantities and reflected in the schedule. The overbreak and design allowance factors are summarized in Table 16-24.

Table 16-24: Waste Rock Development Overbreak and Design Allowance Factors

Item	Value
Overbreak	**10%**
Design Allowance	**20%**

There is no overbreak or design allowance applied to excavations in mineralized resource.

16.3.5 Stoping Methods

There will be both bulk and selective mining methods used to extract the resource.

- Longhole Stoping (Longhole) will be the primary mining method used in greenfield areas.
- Variations of Mechanized Cut and Fill (MCAF) or Drift and Fill (DAF) will be used in areas where Longhole cannot be applied due to local geometry, continuity, or proximity to the historic workings, but the resource is of sufficient grade to support the higher operating cost.

16.3.5.1 LONGHOLE STOPING

A combination of Transverse and Longitudinal Longhole stoping will be used. Generally, due to limited strike length to establish a consistent primary-secondary transverse sequence, even in thicker areas longitudinal mining will be used to reduce the number of stope accesses in waste rock. Sublevels will be established based on the spacing of the existing sublevel elevations where practical, but generally targeting approximately 82 ft (25 m) between sublevels. Production drilling will be primarily down-holes; however up-holes will be used where required to maximize resource recovery.

The design assumptions for Longhole stopes are summarized in Table 16-25.

Table 16-25: Longhole Stope Design Assumptions

Item	Value
Transverse Longitudinal Maximum Blind Uphole Length	Width (HW to FW) > 50 ft (15.0 m) Width (HW to FW) < 50 ft (15.0 m) 50 ft (15.0 m)
Sublevel Interval	Approximately 82 ft (25.0 m) (floor to floor)
Stope Width (along strike)	50 ft (15.0 m)
Drill Hole Diameter Ring Burden Hole Spacing	Top Hammer Electric-Hydraulic 3 inches 6.6 ft (2.0 m) 7.2 ft (2.2 m)
Slot Raise	Drop Raise / Inverse Drop Raise
Explosive Type Detonators Loading Method	Bulk Emulsion Non-Electric Mobile Explosives Loader
LHD	18T Class

Backfill	Cemented Rockfill (CRF) Unconsolidated Rockfill (RF)
Haul	40T Class Haul Truck

Longhole Stope Drilling and Blasting

Production drilling will be completed using electric-hydraulic top-hammer drills drilling 3-inch diameter holes on a 6.6-ft ring burden and 7.2-ft hole spacing. There will be some fan drilling required. Production holes will be loaded with bulk emulsion and detonated with non-electric detonators. A mobile explosive loader will be used to load the holes.

Longhole Stope Production Mucking

Blasted material will be mucked from stopes using an 18T class LHD. When the stope brow is closed, the LHD will be operated with the operator in the cab. When the stope brow is open, the LHD will be operated on remote control with the operator stationed at a remote stand, located a safe distance from the brow and away from the path of the moving LHD.

On levels with a muck-pass dump (generally below 2000L), the LHD will tram and dump directly into the pass. Otherwise, the LHD will tram and dump into a remuck bay typically located near the sublevel access. When a haul truck is present at the remuck, the LHD will load the truck. The height of the drift at the truck loading area will accommodate the truck loading.

16.3.5.2 MECHANIZED CUT AND FILL / DRIFT AND FILL

There is mineralized resource material near the historic mined out stopes. Due to uncertainty of the backfill status of historic stopes and potential for inaccuracy of the survey in the mine model, a selective mining method has been considered. The selective mining method reflects a higher mining cost and mitigates some of the risk associated with the complexity that mining around the existing working presents. A combination of Mechanized Cut and Fill and Drift and Fill (collectively referred to as cut and fill) will be used to recover resource material that is in near proximity to the historic stopes. A 16.4-ft pillar/standoff distance from historic stope has been maintained in the mine design and a systematic probe-hole program to test for the voids will form part of the mining process.

Below 2000L, where the historic primary stopes were backfilled with consolidated sandfill, the cut-and-fill shapes contained in the 16.4-ft pillars were evaluated to determine the potential for extraction. It was determined that approximately 20% was a reasonable assumption (50% of stopes were primaries x 40% extraction of shapes adjacent to primaries) for recovering this material and has been included in the mine plan.

The design assumptions for MCAF and DAF are summarized in Table 16-26.

Table 16-26: MCAF and DAF Design Assumptions

Item	Value
MCAF DAF	Stope Width </= 26.2 ft (8.0 m) Stope Width > 26.2 ft (8.0 m)
Cut Height	16.4 ft (5.0 m)
Cut/Drift Width	Min 13.1 ft (4.0 m) Max 26.2 ft (8.0 m)
Design Gradient	0%
Cuts per Level	5
Level Spacing	As required but typically 82.0 ft (25.0 m)
Attack Ramp Max Gradient	+/-15%
Backfill	MCAF – RF DAF – CRF/RF

16.3.5.3 TRUCK HAULAGE

Hauling Mineralized Resource

For sublevels that do not have access to a muck pass system there will be a truck loading area located near the sublevel access. Above 2000L, the truck will be loaded by the LHD and will haul to the surface pad near the portal. Below 2000L, the truck will haul and dump into the nearest muck pass dump to 4000L. It is anticipated 50T class underground haul trucks will be used for production.

Hauling Waste Rock

Before stope production starts all waste rock will be hauled to surface to a waste rock dump near the portal. Once stope production begins, waste rock will be hauled and dumped at other levels for use as backfill. It is anticipated 40T underground haul trucks will be used for hauling waste rock.

16.3.5.4 BACKFILL

Stopes that will be mined against will be backfilled with cemented rockfill (CRF), and stopes that will not be mined against will be backfilled with unconsolidated rockfill or left open. The CRF will be a blend of mine waste rock and cement slurry. It is assumed that a surface slurry mix plant and boreholes will deliver slurry to the level where CRF is being placed. Haul trucks will deliver waste rock to the backfilling level where an LHD will rehandle the material for dumping into the stope. When CRF is being placed, a metered volume of slurry will be added to the LHD bucket. The LHD will initially dump over a bumper block, but when the backfill reaches the bench the LHD will dump and spread backfill using remote control.

Four percent Normal Portland cement binder content is generally adequate for vertically self-supporting backfill at the given stope dimensions. The backfill assumptions are summarized in Table 16-27.

Table 16-27: Backfill Assumptions

Item	Assumption
Backfill tons to Stope Tons Replacement Factor	50%
Cemented Rockfill Binder Content	4%

16.3.6 Dilution and Mining Recovery

16.3.6.1 DILUTION

There will be Planned Internal Dilution and Unplanned External Dilution.

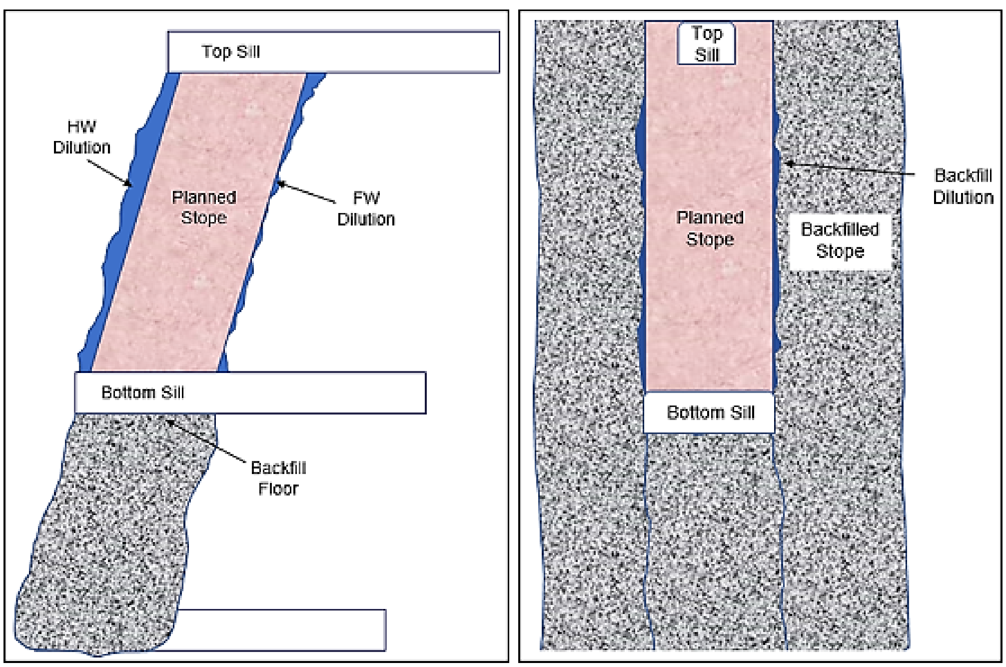
Planned Internal Dilution

Planned Internal Dilution is low grade material and/or waste rock that is contained inside the stope shape and the sill development shape. The Planned Internal Dilution is reported along with the in-situ resource from the Deswik model.

Unplanned External Dilution

Unplanned External Dilution in Longhole stopes will be waste rock from the hangingwall (HW) and footwall (FW) that will overbreak into the stope and will be mined with the stope resource. Additional dilution will come from backfill in adjacent stopes. The sources of unplanned external dilution in Longhole stopes are shown in Figure 16-31.

Figure 16-31: Sources of External Dilution



The estimated Unplanned External Dilution for each mining method will be calculated in the Deswik model and summarized in Table 16-28.

Table 16-28: Unplanned External Dilution Assumptions

Item	Primary Stope Overbreak	Secondary Stope Overbreak	Longitudinal Stope Overbreak
Hangingwall	2 ft (0.6 m)	2 ft (0.6 m)	2 ft (0.6 m)
Footwall	1 ft (0.3 m)	1 ft (0.3 m)	1 ft (0.3 m)
Backfill (from adjacent stopes)	0	1 ft (0.3 m) from each wall	1 ft (0.3 m)
Backfill from mucking floor*	0	0	0

* A slurry cap will be placed to prevent mucking the floor.

16.3.6.2 MINING RECOVERY

A mining recovery factor will be applied to account for stope resource that is planned to be mined but due to losses in the mining process will not be delivered to surface. The mining recovery factor by mining method is summarized in Table 16-29.

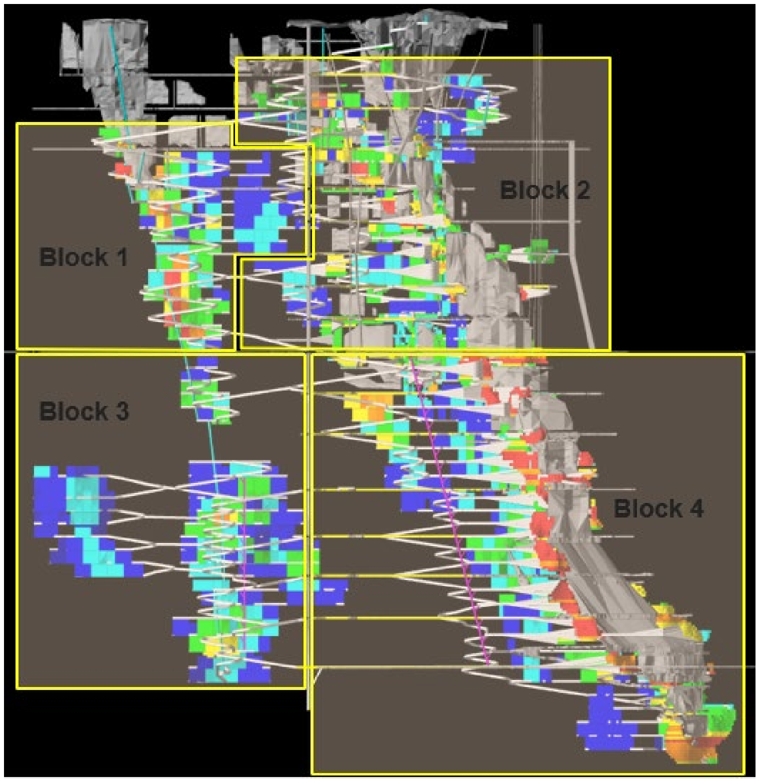
Table 16-29: Mining Recovery Factors

Mining Method	Mining Recovery Factor
Longhole	90%
MCAF / DAF	95%

16.3.7 Mine Quantities and Scheduling

The underground mine has been divided into four main blocks. The blocks are generally defined laterally by the shaft and vertically by the 2000L as shown in Figure 16-32.

Figure 16-32: Mining Blocks



Block 1 and Block 3 are primarily greenfield production areas with longitudinal longhole as the primary mining method. Block 2 and Block 4 are brownfield production areas with cut and fill used near historic workings and longhole used in areas not influenced by previous mining.

16.3.7.1 PHASES OF UNDERGROUND ACTIVITY

There will be three main phases of underground activity, as follows.

- Advanced Exploration Program
- Phase 1 Production to 2000L
- Phase 2 Production to 4000L

Advanced Exploration Program

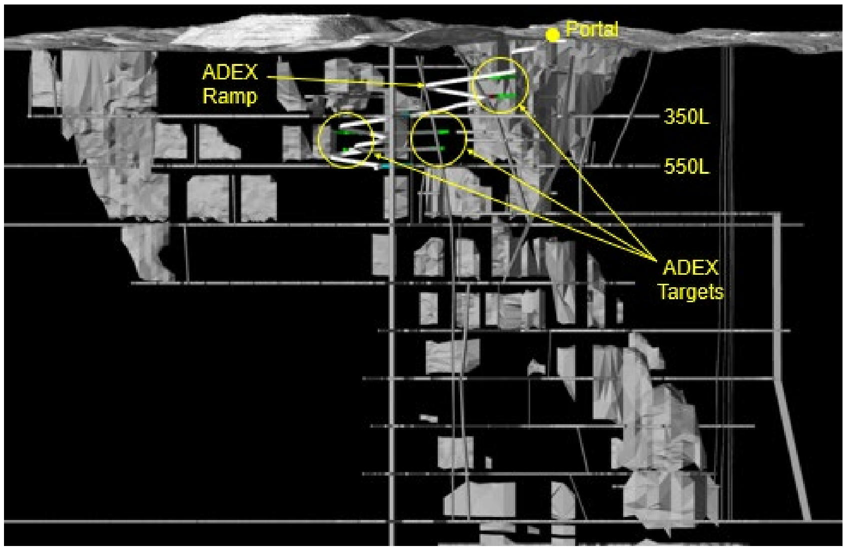
There will be an initial Advanced Exploration (ADEX) program that will include.

- Establishing some of the site surface infrastructure.
- Dewatering flooded underground workings to 550L from surface via the shaft.
- Developing an Access Ramp to 550L elevation.
- Mining an approximately 31,200-ton underground bulk sample from targeted mineralized zones.
  - 10,100 tons from the 101 Zone
  - 10,200 tons from the 109 FW Zone

- 10,900 tons from the Remnant Zone
- Diamond drilling from underground locations.

The mine design for the ADEX program is shown in Figure 16-33.

Figure 16-33: ADEX Mine Design



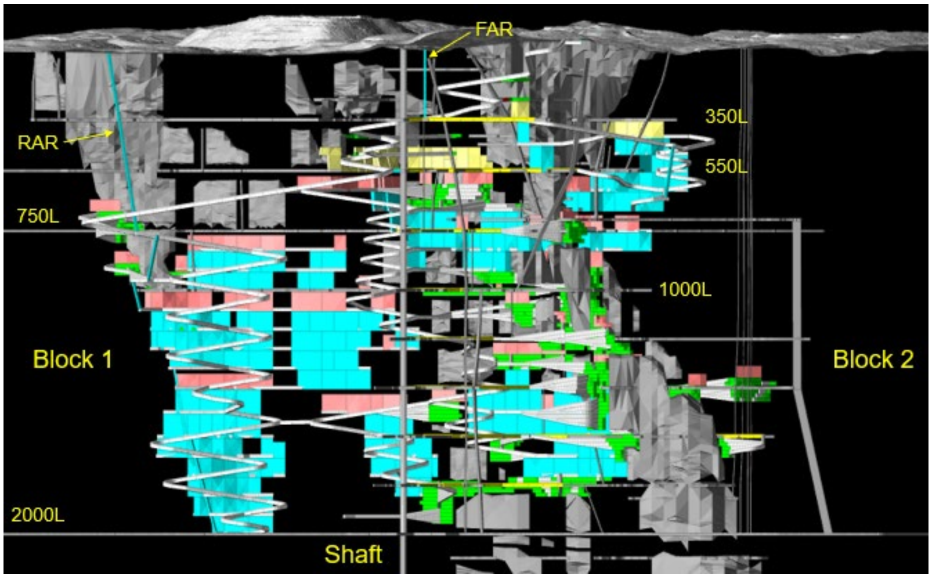
Production to 2000L

Following ADEX, the Project capital period for Phase 1 will start to prepare for production. Phase 1 production will be from Block 1 and Block 2 using the Access Ramp for material and personnel movement, with a second egress established in the existing shaft. During Phase 1, the production shaft facilities will be prepared for producing from Block 3 and Block 4 during Phase 2. The following activities will occur during Phase 1.

- Expand surface site infrastructure to support increased underground activity.
- Continue dewatering the flooded underground workings and establish egress in the shaft in stages, initially to 550L and then to 1000L and 2000L.
- Develop a return air raise and fresh air raises to surface and establish internal ventilation raises.
- Install surface ventilation plants to increase ventilation flows to support additional mobile equipment and mining activity.
- Continue ramp and lateral development in Block 1 and Block 2.
- Expand underground systems (power, communications, dewatering, process water, compressed air).
- Expand underground infrastructure.
- Start longhole and cut and fill production using trucks to haul muck to surface via the ramp.
- Recondition, re-equip, and commission the shaft and loading pocket and establish the hoisting facility in preparation for Phase 2.
- Start the ramp, lateral, and vertical development into Block 3 and Block 4 to prepare for production.

The mine design for Phase 1 is shown in Figure 16-34.

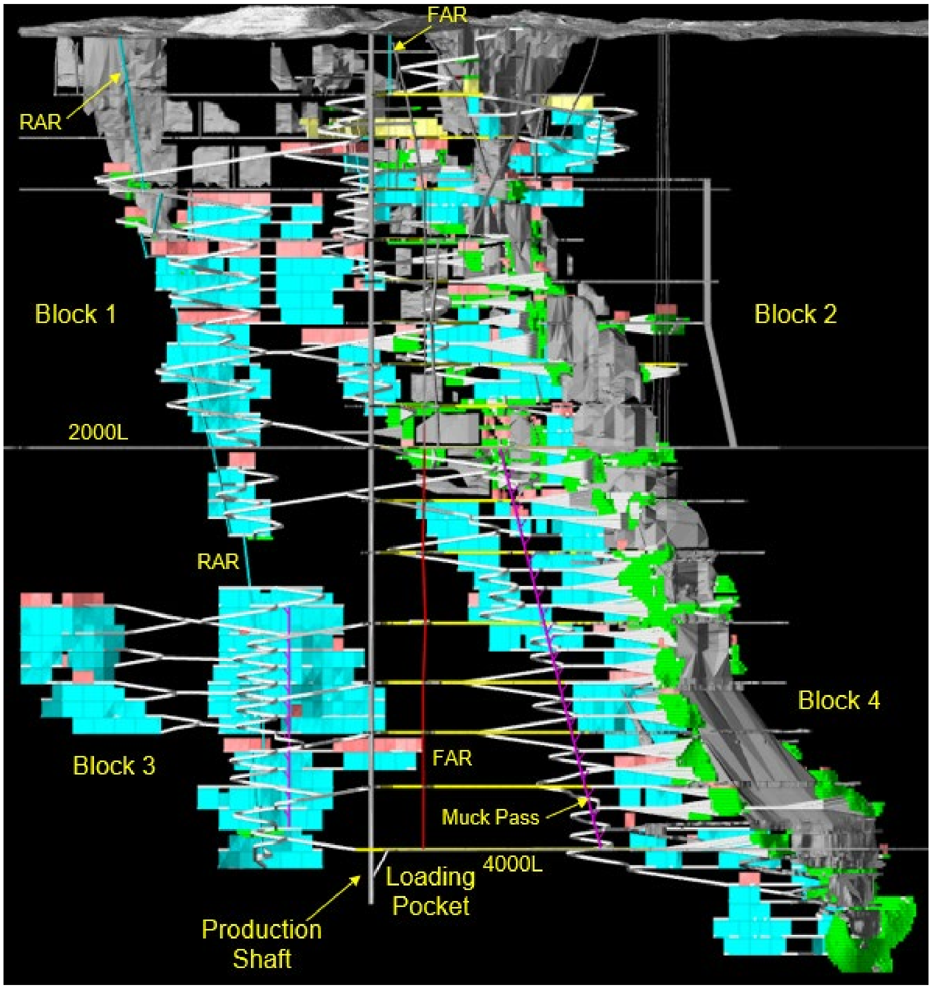
Figure 16-34: Phase 1 Mine Design



Phase 2 – Production to 4000L

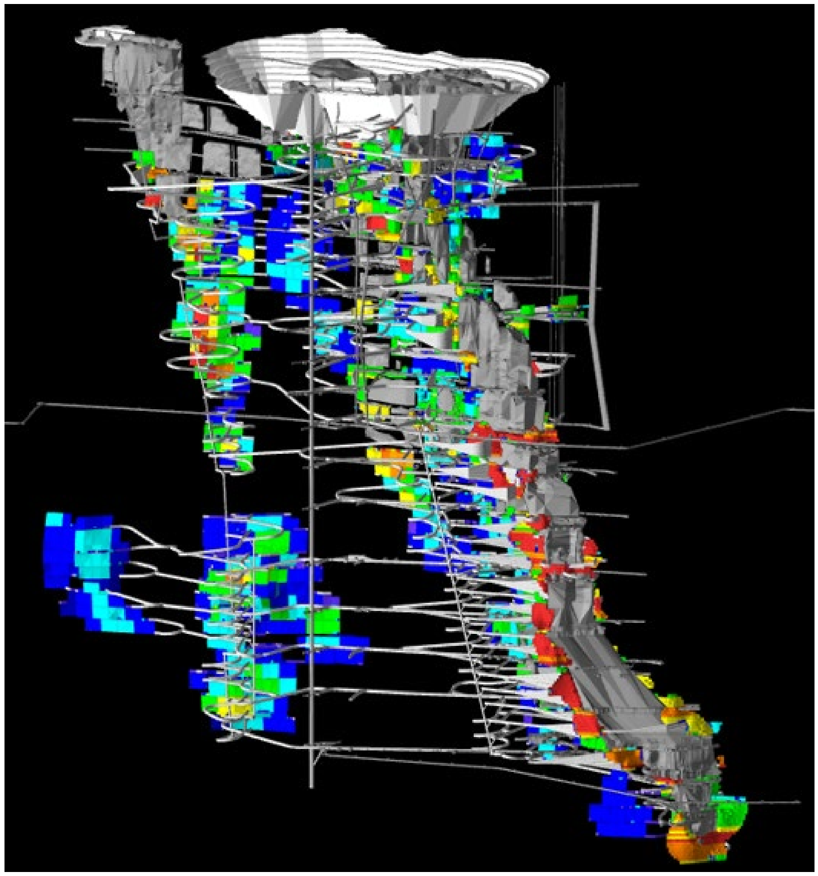
Phase 2 will be an expansion of the mine below 2000L, down to approximately 4500L, using the shaft as the main facility for personnel and material handling. During Phase 2 there will be continued mining above 2000L as production from Block 3 and Block 4 ramps up. The production from Phase 2 will be timed to ramp up as production from Phase 1 starts to ramp down to avoid a drop in the production profile. The mine design at the end of Phase 2 is shown in Figure 16-35.

Figure 16-35: Underground Mine Design at the End of Phase 2



The final combined Open Pit and Underground Mine Design at the end of the mine life is shown in an isometric view in Figure 16-36.

Figure 16-36: Isometric View of Final Combined Open Pit and UG Mine Design



16.3.7.2 DEVELOPMENT QUANTITIES

The lateral and vertical development quantities with allowance and overbreak factors applied are summarized by ADEX and mining block in Table 16-30.

Table 16-30: Lateral and Vertical Development Quantities

Item	ADEX (feet)	Block 1 (feet)	Block 2 (feet)	Block 3 (feet)	Block 4 (feet)	Total (feet)
Ramp	4,629	11,875	16,310	20,915	24,224	77,952
Lateral Waste/Crosscuts	1,909	6,859	8,341	10,044	21,752	48,905
Rehab Existing Drifts	0	0	9,375	0	12,378	21,753
Stope Sills	1,701	12,339	16,003	25,056	23,786	78,884
Attack Ramp (cut and fill)	0	3,084	26,560	311	69,192	99,147
Total Lateral Development (ft)	**8,239**	**34,157**	**76,588**	**56,326**	**151,331**	**326,641**
Total Lateral Development (m)	**2,511**	**10,411**	**23,344**	**17,168**	**46,128**	**99,560**
Vertical Development (ft)	**0**	**2,028**	**552**	**3,085**	**1,982**	**7,648**
Vertical Development (m)	**0**	**618**	**168**	**940**	**604**	**2,331**
Total Waste Rock (tons)	**186,738**	**588,923**	**1,495,614**	**827,268**	**3,100,529**	**6,199,072**
Total Waste rock (tonnes)	**169,406**	**534,262**	**1,356,798**	**750,485**	**2,812,753**	**5,623,703**

16.3.7.3 DEVELOPMENT ADVANCE RATE

To estimate the development advance rate the development cycle was broken into each component of the drill, blast, muck, and support cycle. The estimated advance rates for each Jumbo in the fleet used in the Deswik schedule are summarized in Table 16-31.

Table 16-31: Development Advance Rates

Scenario	Advance Rate (feet per day)	Notes
Single Face Ramp	14.3 ft/d (4.35 m/d)	During initial ADEX Ramp development after an initial learning curve period. Blast at will during ADEX.
Multiple Face (Ramp and other Lateral)	28.6 ft/d (8.70 m/d)	Two or more faces available to the Jumbo. Maximum 14.3 feet per day for a face.

16.3.7.4 DEVELOPMENT PROFILE

Advanced Exploration Program

Following construction of the portal and the first stages of dewatering flooded workings, Access Ramp development for ADEX will begin with a mining contractor. After an initial learning curve, the estimated Access Ramp advance will be 14.3 ft per day (4.35 m per day). Once the first bulk sample target area is reached there will be additional working faces for the jumbo and the advance rate will increase to a maximum of 28.6 ft per day (8.70 m per day), with a constraint that no face can advance at more than 14.3 ft per day (4.35 m per day). There will be a tie-in to the existing near shaft development on 550L to allow a ventilation circuit to be established with the shaft as a return air route for Phase 1. There will be a single development crew (i.e., one jumbo operating) during the ADEX program.

Phase 1

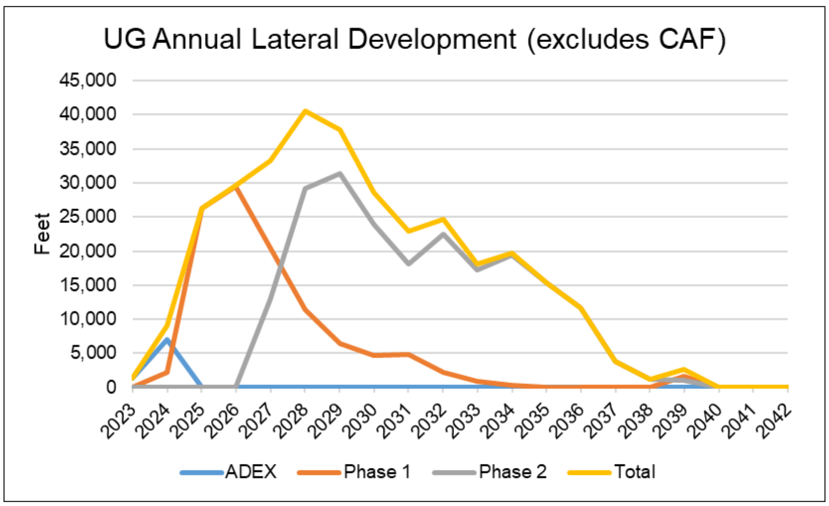
The development and construction completed during ADEX will set-up the mine up with egress and infrastructure to increase the ventilation capacity to support additional development equipment. The priority at the start of Phase 1 will be to develop a return air raise to surface and a short fresh air raise near the shaft. As the ventilation infrastructure come available, three additional Jumbos will be phased-in to complete Block 1 and Block 2 ramp and lateral development, longhole stope sills, and cut and fill mining.

Phase 2

When development activities ramp down in Block 1 and Block 2, the jumbos will continue below 2000L and into Block 3 and Block 4 to start production from new zones. There will also be off-shaft development from lower shaft stations from the shaft to prepare the lower mining horizons.

The development profile for ADEX and by mining block is summarized in Figure 16-37.

Figure 16-37: Annual Lateral Development Footage



16.3.7.5 PRODUCTION QUANTITIES

The stope and development shapes are included in the Deswik mine model and schedule. The production quantities have been reported from the Deswik model and include dilution and mining recovery. The diluted and mining recovered production quantities for ADEX and each mining block are summarized in Table 16-32.

Table 16-32: Diluted and Recovered Production Quantities

Production Source	ADEX	Block 1	Block 2	Block 3	Block 4	Total
Development (tons)	21,244	203,566	366,263	419,591	614,439	1,625,103
Longhole Stopes (tons)	0	1,790,524	2,595,443	3,282,108	3,351,941	11,020,016
Cut and Fill Stopes (tons)	0	114,611	1,124,696	9,109	4,045,722	5,294,138
Total Production short tons	**21,244**	**2,108,700**	**4,086,403**	**3,710,808**	**8,012,102**	**17,939,257**
Total Production metric tonnes	**19,272**	**1,912,980**	**3,707,122**	**3,366,388**	**7,268,457**	**16,274,220**
Average NSR / short ton	**$198.61**	**$180.34**	**$159.03**	**$130.57**	**$197.09**	**$172.69**
Average NSR / metric tonne	**$218.93**	**$198.79**	**$175.30**	**$143.93**	**$217.25**	**$190.36**
Average %Ni	0.676	0.519	0.658	0.517	0.736	0.647
Average %Cu	0.709	0.513	0.552	0.395	0.703	0.583
Average %Co	0.020	0.018	0.025	0.018	0.026	0.023
Average Pt (ppm)	1.455	1.281	0.470	0.362	0.632	0.616
Average Pd (ppm)	0.713	1.577	0.380	0.669	0.679	0.715
Average Au (ppm)	0.400	0.773	0.294	0.188	0.410	0.380
Backfill short tons	**0**	**945,735**	**1,846,729**	**1,633,806**	**3,672,303**	**8,098,572**
Backfill metric tonnes	**0**	**857,956**	**1,657,324**	**1,482,164**	**3,331,457**	**7,346,901**

16.3.7.6 PRODUCTION RATES

Longhole Stopes

The longhole stope production rate was estimated from an average stope size of 15,700 tons. The stope productivity was estimated from the drill, blast, muck, backfill cycle days summarized in Table 16-33.

Table 16-33: Longhole Stope Cycle for 15,700-ton Stope

Cycle Component	Days
Drill	17
Blast/Muck	12
Backfill Placement	13
Backfill Cure (CRF)	14

When mining a stope adjacent to a stope filled with CRF, it is assumed the drilling will occur simultaneous with the backfill cure. Longhole stopes will produce at approximately 1,300 tons per day during the Blast/Muck cycle. To maintain an active drawpoint for mucking each shift (1,300 tons per day sustained production from Longhole stopes) there will be 4 to 5 stopes active in various stages of the cycle.

Cut and Fill Stopes

The cut and fill production rate was estimated from the development performance. The Jumbos assigned to cut and fill mining will have multiple attack ramps to work with so when a workplace is in the backfill and attack ramp slashing cycle there is another workplace available in the production cycle. Each cut and fill jumbo will advance 28.6 ft per day (16.4 ft x 16.4 ft drift equivalent) averaging 720 tons per day.

16.3.7.7 PRODUCTION PROFILE

The underground will operate 365 days per year however the production schedule is based on 360 days per year to account for unforeseen disruptions. Mine development and production scheduling were completed using Deswik scheduling software, with the schedule linked to the Deswik mine model. The development and production schedule are based on dependencies linked within the mine model and the assigned development and production resources. The production profile includes a combination of development and stope production. To achieve reasonable stope production the mine was divided into several mining fronts that could be sequenced and mined simultaneously (until the fronts approach one another).

During Phase 1, combined production from Block 1 and Block 2 will ramp up and average approximately 2,400 tons per day. Once the shaft is commissioned for Phase 2 in 2028, overall production will increase to approximately 4,000 tons per day when all blocks are producing before ramping down and ending in 2039. The stopes in Block 2 near the interface with the Open Pit have been delayed until the end of mine life.

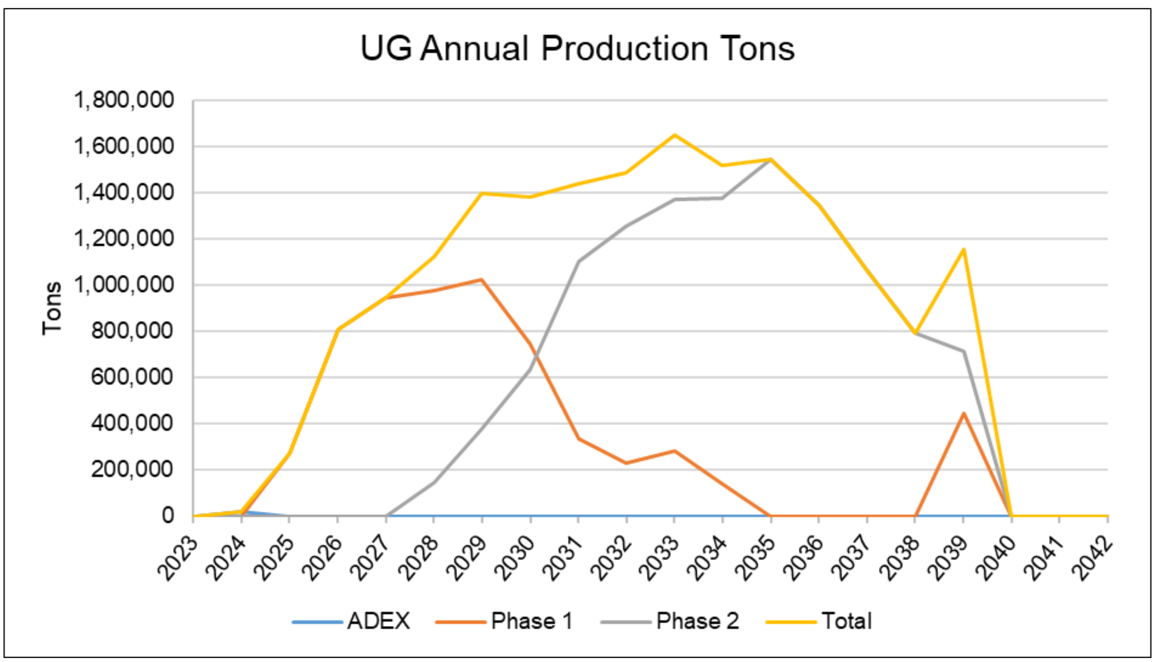
The underground production tonnage and average NSR/grade profile (including stope sill development) is summarized in Table 16-34. The annual tonnage profile is shown graphically in Figure 16-38.

Table 16-34: Annual Production Tons, NSR, and Grade Profile

Item	2024	2025	2026	2027	2028	2029	2030
Short Tons Metric Tonnes	21,244 19,273	273,458 248,077	806,611 731,745	942,886 855,371	1,124,764 1,020,369	1,397,480 1,267,773	1,379,258 1,251,242
NSR per short ton	$198.61	$138.88	$160.60	$163.74	$174.72	$169.45	$179.62
%Ni	0.676	0.526	0.604	0.556	0.583	0.600	0.656
%Cu	0.709	0.486	0.513	0.567	0.524	0.562	0.598
%Co	0.020	0.020	0.022	0.021	0.021	0.021	0.023
Pt (ppm)	1.455	0.839	0.662	0.644	1.036	0.859	0.661
Pd (ppm)	0.713	0.566	0.742	0.896	1.167	0.940	0.798
Au (ppm)	0.400	0.334	0.358	0.506	0.557	0.525	0.407
Item	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**
Short Tons Metric Tonnes	1,439,258 1,306,200	1,484,791 1,346,980	1,650,235 1,497,068	1,516,953 1,376,157	1,543,573 1,400,306	1,346,399 1,221,432	1,063,595 964,877
NSR per short ton	$210.00	$173.54	$166.27	$170.43	$176.73	$170.13	$173.29
%Ni	0.792	0.663	0.648	0.661	0.681	0.666	0.650

Item	2024	2025	2026	2027	2028	2029	2030
%Cu	0.751	0.550	0.571	0.562	0.590	0.617	0.605
%Co	0.027	0.024	0.023	0.023	0.023	0.022	0.023
Pt (ppm)	0.617	0.564	0.493	0.529	0.551	0.473	0.579
Pd (ppm)	0.760	0.682	0.577	0.635	0.633	0.546	0.637
Au (ppm)	0.413	0.363	0.304	0.320	0.361	0.305	0.332
Item	**2038**	**2039**	**Total**				
Short Tons Metric Tonnes	791,349 717,899	1,156,823 1,049,452	17,939,257 16,274,220				
NSR per short ton	$153.75	$164.26	$172.69				
%Ni	0.574	0.657	0.648				
%Cu	0.519	0.581	0.583				
%Co	0.021	0.023	0.023				
Pt (ppm)	0.457	0.496	0.616				
Pd (ppm)	0.645	0.485	0.715				
Au (ppm)	0.328	0.286	0.380				

Figure 16-38: Production Short Tons Profile



16.3.8 Mobile Equipment

The mobile equipment fleet will include units that are commonly used in similar development and production applications. The estimated peak mobile equipment fleet during each phase of activity are summarized in Table 16-35.

Table 16-35: Mobile Equipment Fleet

Item	ADEX (No.)	Phase 1 (No.)	Phase 2 (No.)
Development / Cut and Fill			
Jumbo	1	4	5
Explosives Loader	0	1	1
LHD (14T class)	1	4	5
Mechanical Bolter	1	5	6
Longhole Stope			
Longhole Drill	0	3	3
Explosives Loader	0	2	2
LHD (18T class)	0	2	2
Truck Haulage			
Haul Truck (40T class)	2	2	2
Haul Truck (50T class)	0	4	2
Backfill Placement			
LHD (18T class)	0	2	2
Services and Construction			
Cable Bolter	0	1	1
Service LHD	0	1	1
Boom Truck	0	1	1
Scissor Lift	0	2	2
Grader	0	1	1
Mechanic's Truck	1	1	1
Electrician's Truck	0	1	1
Fuel/Lube Truck	0	1	1
Personnel Carrier	0	2	2
Light Vehicles (Jeep, Tractor)	0	8	10
Surface Mobile Equipment			
Pickup Truck	1	2	2
Front-End Loader	0	1	1

16.3.9 Mine Labour

The underground mine will operate 2 x 10.5-hour shifts per day. There will be four crew rotations required to support underground operations. For example, crews A and B will be on-site working Dayshift and Nightshift while crews C and D are on days off. The ADEX program and capital project period during Phase 1 will be completed by contractor personnel. Once production begins during Phase 1, there will be a transition from contractor personnel to Owner personnel.

The estimated peak Owner personnel on payroll during ADEX, Phase 1, and Phase 2 to support the underground operation are summarized in Table 16-36. After the initial Phase 1 project period, it is anticipated that contractors will continue to complete specialized work such as raising, shaft reconditioning, and diamond drilling over the life of mine.

Table 16-36: UG Mine Owner's Labour (on Payroll)

Position	ADEX (No.)	Phase 1 (No.)	Phase 2 (No.)
Management			
Mine Manager	0	1	1
HR / Health / Safety Superintendent	0	1	1
UG Mine Superintendent	1	1	1
UG Maintenance Superintendent	0	1	1
Mine Technical Services			
Environment Manager	1	1	1
Manager Mine Engineering	1	1	1
Senior Mine Engineer	0	2	2
Mine Engineer – Ventilation / Industrial Hygiene	0	1	1
Mine Engineer – Rock Mechanics / Backfill	0	1	1
Mine Technicians and Surveyors	0	4	4
Manager Mine Geology	1	1	1
Senior Mine Geologist	0	1	1
Mine Geologist	1	3	3
Geology Technician	0	1	1
Other Staff			
Controller / Cost Accountant	0	1	1
Payroll / HR Administrator	0	1	1
IT and Programmer	0	1	1
Procurement Manager / Buyer	1	1	1
Warehouse Supervisor	0	1	1
Warehouse Shipper/Receiver	0	1	1
Site Nurse / First Aid Attendant	0	1	1
Janitorial / Dry	0	2	2
Mine Operations Staff/Hourly			
UG Mine General Foreman	0	1	1
UG Trainer	0	1	1
Mine Shift Supervisor	0	10	10
Hoist Person	0	0	4
Shaft Leader	0	0	1
Deck/Cage Tender	0	0	4
Development / Cut and Fill Miner	0	48	60
Longhole Driller	0	12	12

Position	ADEX (No.)	Phase 1 (No.)	Phase 2 (No.)
Longhole Blaster	0	4	4
Longhole LHD Operator	0	8	8
Backfill LHD Operator	0	8	8
Haul Truck Operator	0	20	10
Backfill Slurry Operator	0	4	4
Rockbreaker Operator	0	0	4
General UG Labourer / Grader Operator	0	8	8
Construction Miner - Cable Bolt, Shotcrete, etc.	0	8	8
Mine Maintenance Staff/Hourly			
Maintenance Planner	0	1	1
Mechanical General Foreman	0	1	1
Mechanical Supervisor	0	4	4
Diesel Mobile Equipment Mechanic	0	20	20
Fixed Plant Mechanic / Millwright	0	2	2
Hoist Mechanic	0	0	1
Electrical General Foreman	0	1	1
Electrical Supervisor	0	4	4
Electrician	0	12	12
Instrumentation Technician	0	1	1

16.3.10 Ventilation

The ventilation system will be installed in stages as underground activity progresses from ADEX, to Phase 1, and to the final configuration in Phase 2.

16.3.10.1 ADEX VENTILATION

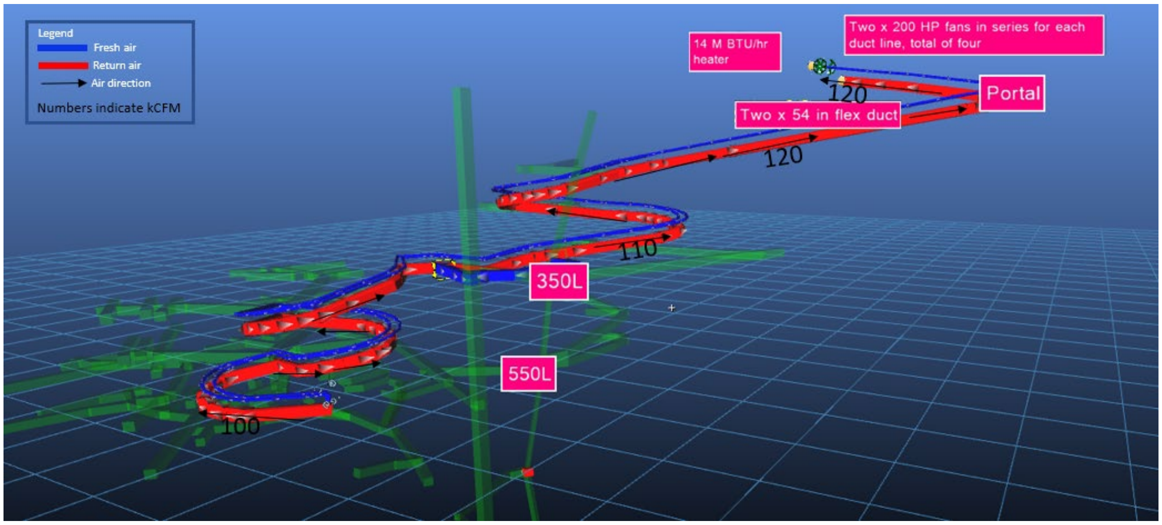
ADEX Stage 1

Ventilation for the ADEX program will have fans and mine air heaters installed at the portal to force approximately 120,000 cubic feet per minute (cfm) of fresh air through ducting to the working face. The initial ventilation set-up is anticipated to include the following.

- Two lines of 56-inch flexible ducting, each with 2 x 200 hp fans installed in series on surface at the portal (four fans total). The fans can be phased in as the ramp face gets further from the portal.
- 14 million BTU per hour propane mine air heater and propane tanks and distribution system.

The ventilation configuration at the end of the ADEX program is shown in Figure 16-39.

Figure 16-39: ADEX Ventilation Schematic



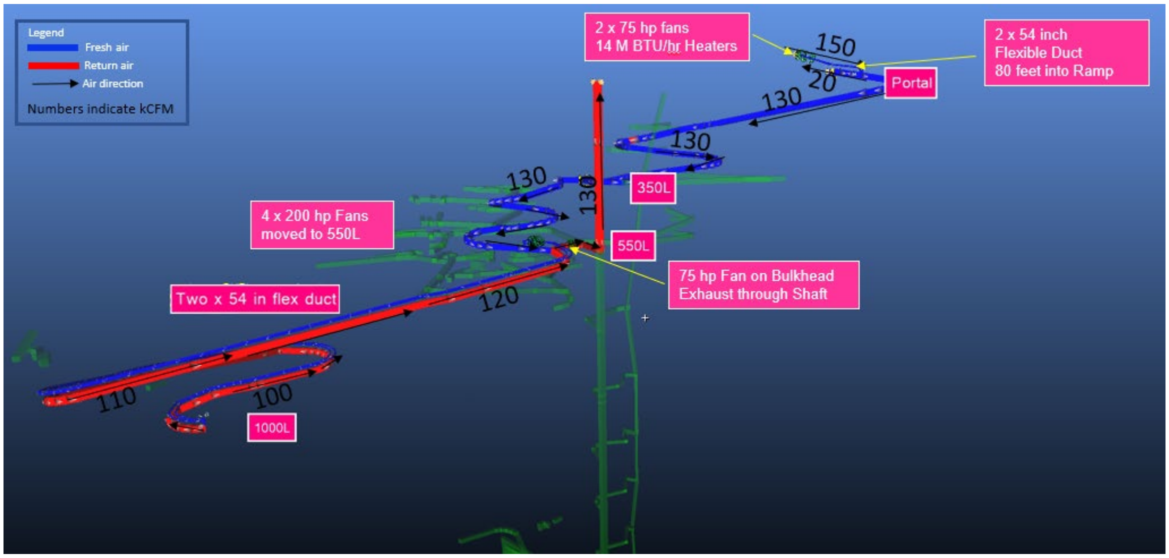
16.3.10.2 PHASE 1 VENTILATION

At the start of Phase 1, the Access Ramp will connect to existing mine workings near the shaft station on 550L and a ventilation bulkhead constructed with an exhaust fan installed to push air up the shaft to create a ventilation loop. The 200 hp fans will be moved from the portal to the ramp near 550L and the two lines of duct will extend continue development to 1000L. The mine air heaters will continue to be located at the portal and there will be small fans at the portal to introduce heated air to the ramp during the heating season. The ventilation system at the start of Phase 1 is anticipated to include the following.

- Portal - 2 x 75 hp fans and 14 million BTU/hr mine air heater.
- At 550L – Connection to the 550L mine workings with a bulkhead and 75 hp exhaust fan to create a ventilation loop and exhaust up the shaft.
- The 550L Ramp - Two lines of 56-inch flexible ducting, each with 2 x 200 hp fans installed in series (four fans total).
- Secondary egress established in the shaft at 550L.

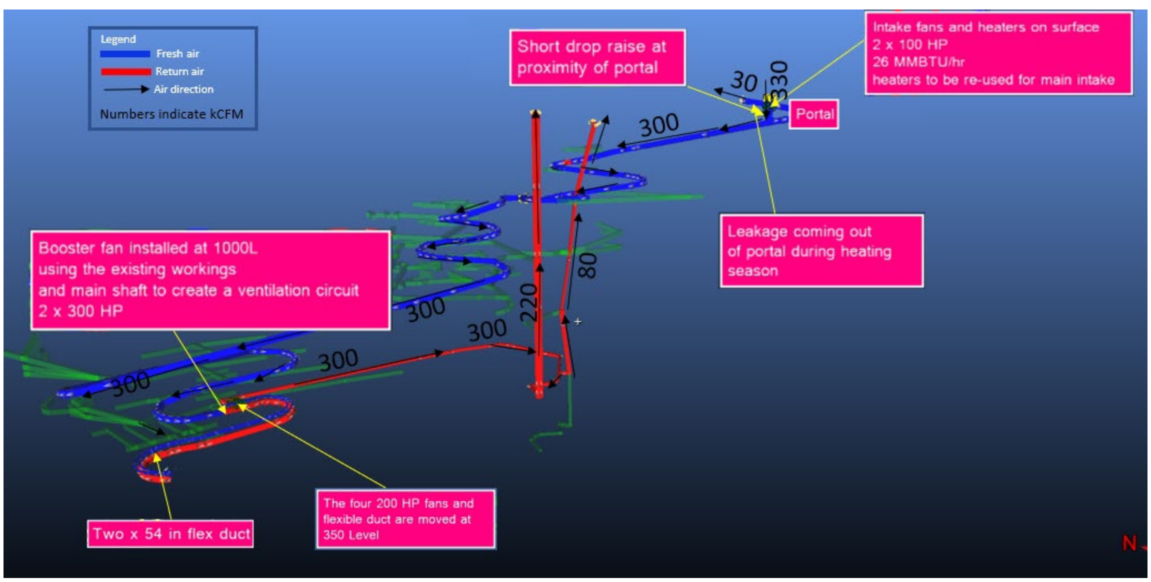
The initial Phase 1 ventilation set-up to continue development to 1000L is shown in Figure 16-40.

Figure 16-40: Phase 1 Initial Ventilation Configuration



Once development reaches 1000L the Access Ramp will connect to existing development to allow the ventilation loop to be advanced to 1000L. Booster fans will be installed on 1000L to increase ventilation flows to 300 kcfm to allow increased underground activity. The shaft will continue to be used for exhaust air. The ventilation configuration once the Access Ramp reaches 1000L is shown in Figure 16-41.

Figure 16-41: Phase 1 Ventilation Configuration at 1000L

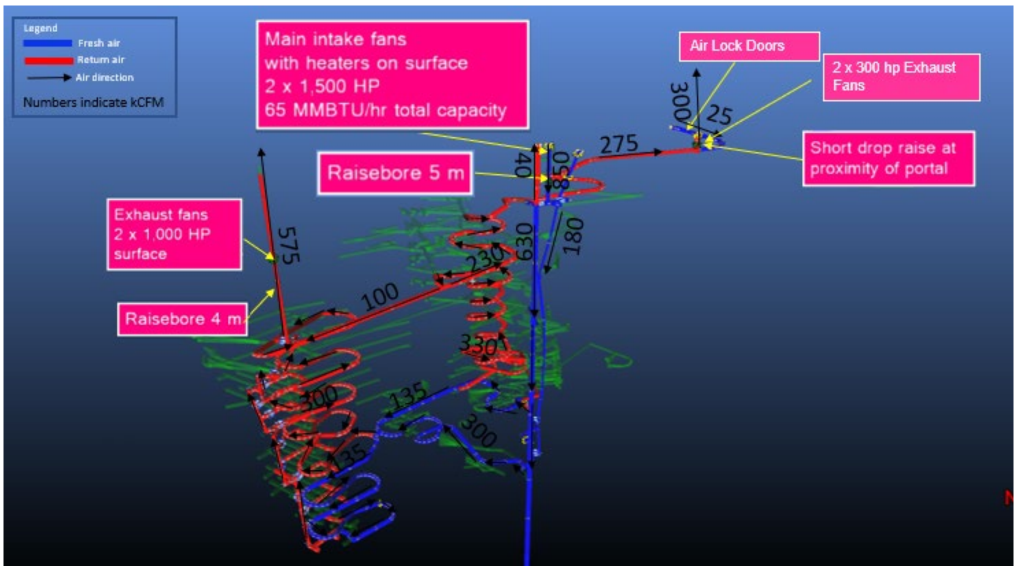


The final ventilation configuration for Phase 1 will include a new 13-ft diameter return air raise (RAR) to surface and a short 16.4-ft diameter fresh air raise (FAR) near the shaft to provide 812 kcfm. The new FAR will connect to the shaft through lateral workings on 350L and the shaft will become the fresh air intake. An existing raise that parallels the shaft will also be used for fresh air intake. The final Phase 1 ventilation configuration is anticipated to include the following.

- Portal – Short raise with 2 x 300 hp exhaust fans.
- Access Ramp – Airlock doors (near portal).
- Main Fresh Air Intake Fan plant with 2 x 1,500 hp fans and 65 million BTU/hour heaters.
- Main Return Air Fan plant with 2 x 1,000 hp fans.
- Internal ventilation raises to distribute air to the levels. Equipped with ladderways for emergency egress.
- Ventilation bulkheads, walls, and regulators to direct air to the working levels.

The final Phase 1 ventilation configuration is shown in Figure 16-42.

Figure 16-42: Phase 1 Final Ventilation Configuration



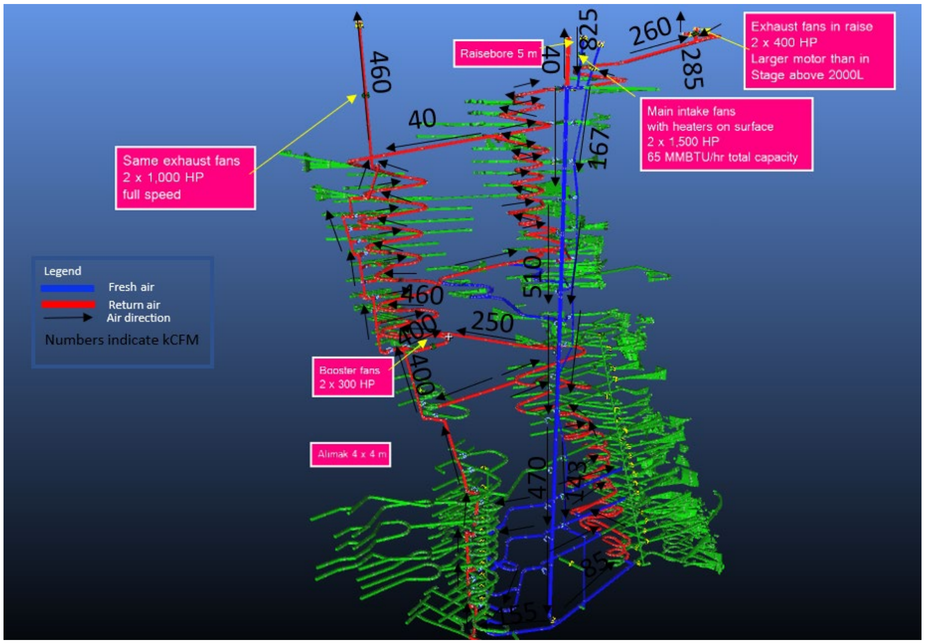
16.3.10.3 PHASE 2 VENTILATION

The ventilation requirements during Phase 2 will be slightly reduced with some truck haulage replaced by hoisting via the shaft, however the surface ventilation plants will not change. The surface Fresh Air plant and Return Air plant will continue to be used during Phase 2. There will be 2 x 300 hp underground booster fans installed on 1000L in the return air system to overcome the increased pressure and the size of the exhaust fans at the portal will be increased. The final Phase 2 ventilation configuration is anticipated to include.

- Portal – Short raise with 2 x 400 hp exhaust fans.
- Access Ramp – Airlock doors (near portal).
- Main Fresh Air Intake Fan plant with 2 x 1,500 hp fans and 65 million BTU/hour heaters.
- Main Return Air Fan plant with 2 x 1,000 hp fans.
- Return Air System – 2 x 300 hp booster fans at 1000L.
- Internal ventilation raises to distribute air to the levels. Equipped with ladderways for emergency egress.
- Ventilation bulkheads, walls, and regulators to direct air to the working levels.

The final Phase 2 ventilation configuration is shown in Figure 16-43.

Figure 16-43: Phase 2 Final Ventilation Configuration



16.3.11 Mine Services

16.3.11.1 COMPRESSED AIR

The equipment requiring underground compressed air is anticipated to include the following.

- Pneumatic drills (jacklegs and stopers)
- Pneumatic pumps
- Blowpipes for cleaning holes
- Shotcrete sprayer
- Refuge Stations
- Maintenance Shop tools
- Alimak Raising

The estimated compressed air consumption for the phases of mine activity is summarized in Table 16-37.

Table 16-37: Compressed Air Consumption

Item	ADEX	Phase 1	Phase 2
Average Consumption	260 cfm	550 cfm	950 cfm
Potential Peak Consumption	865 cfm	1,465 cfm	2,185 cfm

During ADEX and Phase 1 a compressor will be located on a pad near the portal and compressed air delivered to sublevels via a 6-inch pipe in the Access Ramp and internal ramp systems. For Phase 2, there will be a compressed air plant adjacent to the hoist house and compressed air delivered via a 6-inch pipe in the shaft.

16.3.11.2 PROCESS WATER

The underground process water users are anticipated to include the following.

- All drills (jumbo, bolter, longhole, cable bolter, jacklegs/stopers, diamond drill)
- Dust Suppression – Hose/nozzle, water sprays
- Shotcrete
- Wash Bay
- Miscellaneous washing

The estimated process water consumption for the phases of mine activity is summarized in Table 16-38.

Table 16-38: Process Water Consumption

Item	ADEX	Phase 1	Phase 2
Average Consumption	90 USgpm	190 USgpm	245 USgpm
Potential Peak Consumption	145 USgpm	380 USgpm	430 USgpm

A pre-engineered pumphouse building will transfer process water from the water source to water storage tanks. During ADEX and Phase 1 the process water tank will be located on a pad near the portal and process water air delivered to sublevels via a 6-inch pipe in the Access Ramp and internal ramp systems. For Phase 2, there will be a water tank adjacent to the hoist house and process water delivered via a 6-inch pipe in the shaft.

16.3.11.3 DEWATERING SYSTEM

Sources of water inflow will include process water introduced to the mine for drilling, washing, and dust suppression, water inflow captured by openings to surface, and water inflow from the surrounding rock mass. It is understood that the average pumping rate when the mine was in previously in operation was approximately 300 to 400 USgpm and up to 600 USgpm during the spring snow melt season. It is assumed the same pumping capacity will be required when the mine footprint peaks during Phase 2.

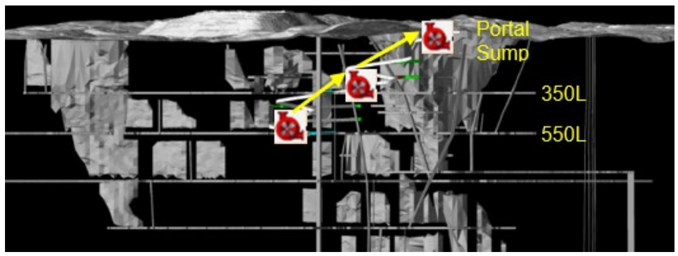
The mine dewatering system is anticipated to include the following.

- Development gradient designed to direct water to flow along a ditch on the floor.
- Collection sumps on each sublevel that will either gravity feed to a sump at a lower elevation via a borehole or be equipped with a submersible pump to transfer water to another sump.
- Intermediate sump/pump stations to transfer water to Main Sump/Pump Stations
- Main Sump/Pump Stations to filter solids and pump clean water to surface.

ADEX Dewatering System

During ADEX, water will be collected in sumps on the bulk sample sublevels. As Access Ramp development advances deeper, water collected in sumps at lower elevations will pump to sumps at higher elevations using submersible pumps. A schematic of the ADEX dewatering system is shown in Figure 16-44.

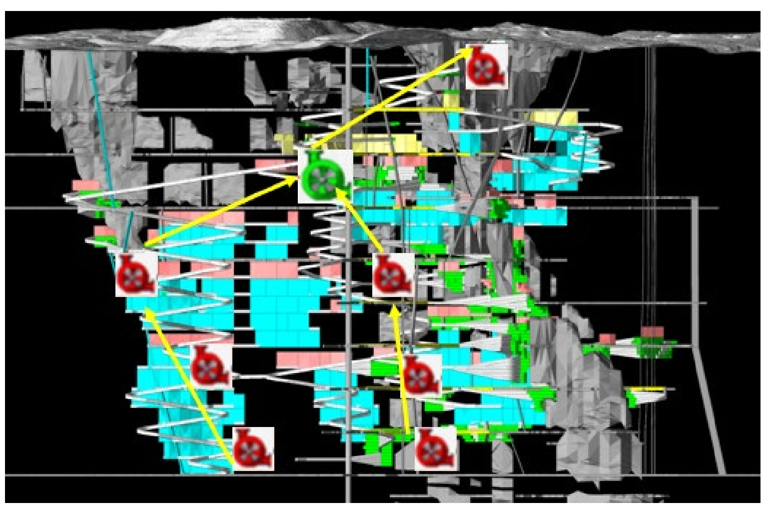
Figure 16-44: ADEX Dewatering System Schematic



Phase 1 Dewatering System

The Phase 1 pumping system will be an expansion of the cascade-up approach established during ADEX. To prevent most catchment water inflow from the openings to surface from entering deeper in the mine, an Intermediate Pump station will be established around the 550L to collect and pump to surface along with other water from sumps below. Water will be collected in sumps on some sublevels and pumped to sumps (using submersible pumps) at higher elevations until eventually collected at the Intermediate Pump Station on 550L and discharged on surface. For sublevels that do not have a submersible pump, water will gravity feed via boreholes to a sump that is part of the cascade-up system. A schematic of the Phase 1 dewatering system is shown in Figure 16-45.

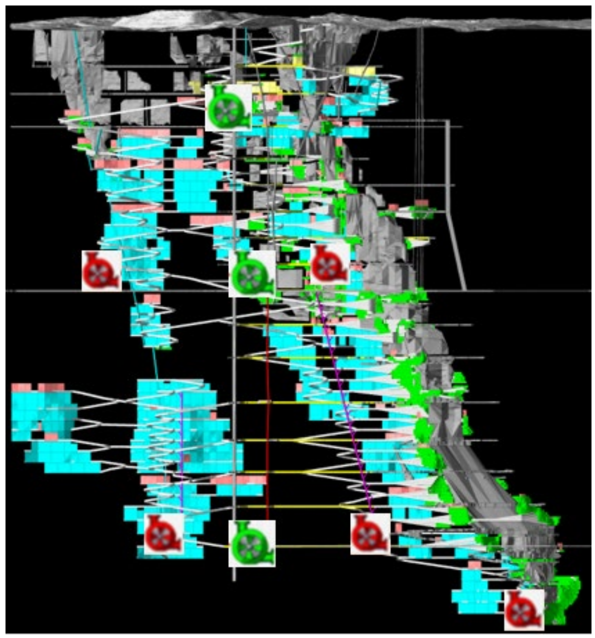
Figure 16-45: Phase 1 Dewatering System Schematic



Phase 2 Dewatering System

The final mine dewatering system will be established during Phase 2. Above 2000L, mining will be largely complete and sumps that continue to collect inflow water will gravity feed to collection sumps on 2000L for transfer to a Main Pump Station near the shaft. Below 2000L, water will gravity feed to collection sumps on 4000L for transfer to a Main Pump Station near the shaft. For sublevels below 4000L, a cascade-up system will transfer water to the Main Pump Station. A schematic of the Phase 2 dewatering system is shown in Figure 16-46.

Figure 16-46: Phase 2 Dewatering System Schematic



16.3.11.4 ELECTRICAL POWER DISTRIBUTION

There is a nearby electrical substation that will be the tie-in point for power to the mine. An overhead powerline will be constructed from this tie-in point to a new site substation. Due to the long lead time for key electrical equipment, it is anticipated the ADEX program will be powered with diesel generators while grid power is being established for Phase 1. A list of power users and load list was developed for each phase of activity to estimate the distribution infrastructure and power consumption. The estimated demand load for ADEX and mining phases are summarized in Table 16-39.

Table 16-39: Power Demand Load for each Phase of Activity

Phase of Activity	Total (MVa)
ADEX	2.2
Phase 1	9.1
Phase 2	14.2

ADEX Power Distribution

The estimated power requirements will be approximately 2.2 MVa for the ADEX program. The power distribution during ADEX will consist of the following main infrastructure.

- Diesel generators at the future location of the site substation, and overhead distribution to the surface facilities and to the underground feeder at the portal.
- Main power cables suspended from the back of the Access Ramp, with distribution onto the sublevels.
- An underground switchroom with a Mine Power Center (MPC) and associated equipment.
- Electrical cut-outs on each sublevel to house panels, starters, and communications equipment.

Phase 1 Power Requirements

The estimated power requirements will be approximately 9.1 MVa for Phase 1. The power distribution during Phase 1 will consist of the following main infrastructure added to the ADEX infrastructure.

- Overhead power line from the tine-in point at an existing substation to a new site substation.
- Site substation and overhead distribution to the surface facilities and to the underground feeder at the portal.
- Additional underground switch rooms and MPC's as the mine expands further into Block 1 and into Block 2.
- Additional electrical cut-outs on new sublevels to house panels, starters, and communications equipment.

Phase 2 Power Requirements

The estimated power requirements will be approximately 14.2 MVa for Phase 2. The power distribution during Phase 2 will consist of the following main infrastructure added to the Phase 1 infrastructure.

- Distribution to the shaft hoist house and headframe area.
- Power supply cable in the shaft to feed the lower mine.
- Additional underground switch rooms as the mine expands into Block 3 and into Block 4.
- Additional electrical cut-outs on new sublevels to house panels, starters, and communications equipment.
- Electrical cut-outs on each sublevel to house panels, starters, and communications equipment.

16.3.12 Underground Communication

The communication systems planned for the underground facilities will use proven technologies. The following systems are planned.

- Fiber optic backbone
- Underground wireless network (LTE)
- Leaky feeder system (radio)
- Blasting system

Fiber Optic Backbone

There will be underground fibre optic backbone connecting to key electrical switch rooms and installations.

Underground Wireless Network (Long Term Evolution – LTE)

The underground wireless network will be LTE using dedicated fibres from the fibre optic backbone to connect hardware throughout the mine. The LTE network will be the primary communication system underground to be used with telephones, tablets, and other peripheral devices. The LTE network will also be used for data collection such as mobile equipment telemetry, equipment/personnel tracking, tele-remote operation, blasting system, and video system.

Leaky Feeder System (Radio Communication)

The leaky feeder system will be the primary method of communication in the ramp during development activities until the fibre optic backbone and LTE system is installed.

Underground, the leaky feeder system will be routed along the main ramp and will split onto levels.

Blasting System

A central blasting system will be used to for blast initiation.

16.3.13 Underground Infrastructure

16.3.13.1 REFUGE STATIONS

There will be permanent Refuge Stations at strategic locations where larger numbers of personnel gather frequently (such as the maintenance shop). The permanent refuge stations will also be used as a lunchroom. Portable refuge stations will be located at key areas and near the working face in headings being developed away from the main infrastructure. Portable refuge stations will be used during emergency conditions only.

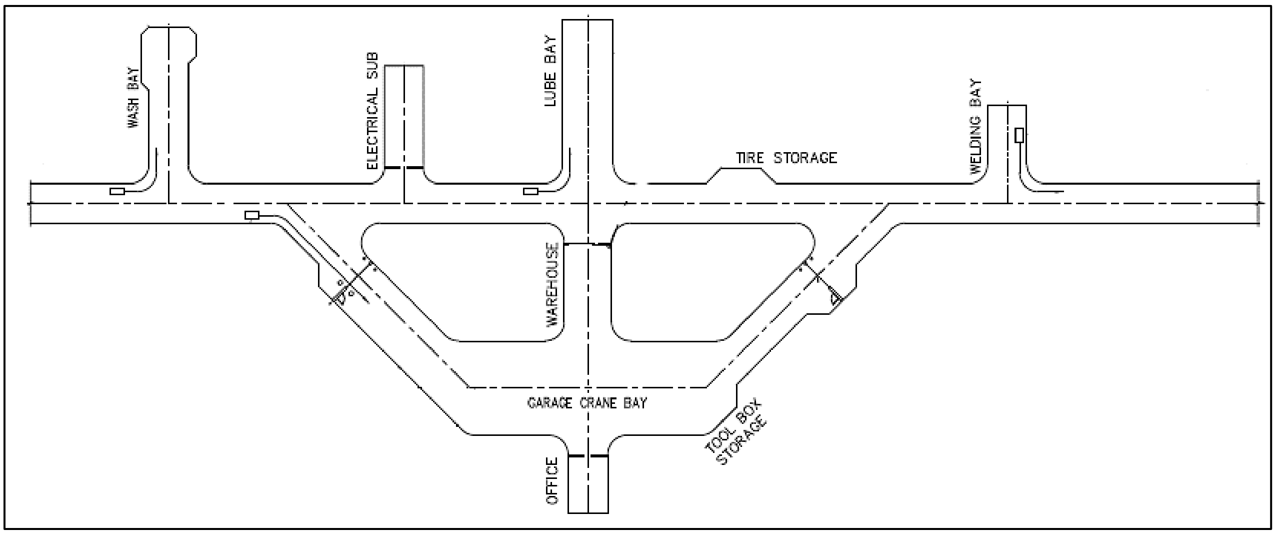
Portable refuge stations will be self-contained manufacturer-supplied and located in repurposed excavations. Each portable refuge station is capable of housing 12 people for 36 hours and will be supplied with oxygen by bottled systems and not through a compressed-air line.

16.3.13.2 MAINTENANCE FACILITIES

During Phase 1 equipment that is easier to tram will be brought to surface for repairs and servicing. For equipment such as jumbos, longhole drills, and mechanical bolters that are more difficult to tram, there will be a single bay satellite maintenance shop for minor repairs and servicing.

During Phase 2, when ramp distances to surface become longer, there will be a multiple bay underground maintenance shop that can service up to four vehicles. The maintenance shop will include two service bays, a crane bay, welding bay, office, hose shop area, electrical equipment room, lubricant storage, and additional storage bays. An example configuration of the maintenance shop area is shown in Figure 16-47.

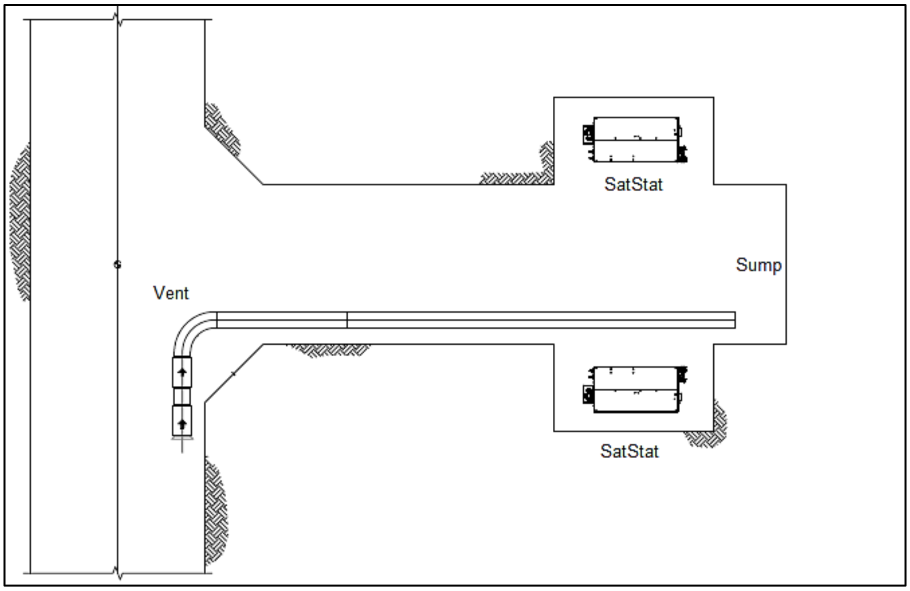
Figure 16-47: Example of Maintenance Shop Area



16.3.13.3 FUEL AND LUBRICANTS

Fuel bays will consist of SatStats® (or equivalent) for fuel and lubes dispensing. Equipment that travels to surface on a routine basis (such as Haul Trucks, Boom Truck, and Personnel Carriers) will be fuelled on surface. Fuel and lubes will be delivered by a fuel truck. The fuel truck will also deliver fuel to equipment such as Jumbos. The typical arrangement of the SatStat® fuel bay is shown in Figure 16-48. It is anticipated there will be three fuel bays for Phase 2.

Figure 16-48: Typical Fuel Bay Arrangement



16.3.13.4 EXPLOSIVES AND DETONATORS

The mine is close to the extensive mine supply network offered in the Sudbury area. It has been assumed that the delivery of explosives and detonators will be frequent and only small storage bays with no large inventory will be established underground. The explosives supplier will deliver explosives and detonators and other blasting accessories to the material staging area at the portal or shaft for subsequent delivery underground by mine services personnel and equipment. Prior to establishing the underground storage bays, explosives and detonators will be stored in magazine on surface.

16.3.13.5 MATERIAL DELIVERY

During ADEX and Phase 1 material and consumables such as ground support, piping, ventilation ducting, etc. will be delivered to a staging area on the pad near the portals for subsequent delivery to underground storage bays. Boom trucks and service LHDs equipped with fork attachments will be used to transport material.

During Phase 2, most materials will be delivered via the shaft cage compartment, while larger/bulkier materials will continue to be delivered via the ramp.

16.3.13.6 PERSONNEL MOVEMENT

During ADEX and Phase 1, underground workers will prepare for the shift on surface and will be transported underground in personnel carriers or on assigned equipment. At the end of shift, underground workers will board the personnel carriers at staging areas and transported to surface. During Phase 1, there will be larger bus-style personnel carriers (24-person capacity) as well as smaller light vehicles. During Phase 2, the service cage in the shaft will be the primary means to move personnel with the ramp used for work in the upper mine or when the cage is not available.

17 RECOVERY METHODS

The recovery approach discussed in this report is based on an ore sale agreement with a local mining company. Detailed metallurgical testing of the 109 FW, and general knowledge of the behaviour of contact deposits within the Sudbury Basin, indicate that processing of these resources will yield positive returns. Based on initial discussions with a local mining company, any extracted material from the Crean Hill property would be processed together with its other feeds.

Before entering into an ore sale agreement with a local mining company, mineralogical and metallurgical testing will be performed on samples that represent the materials that will be processed, according to the local mining company's selection and testing protocols. The test results will be conducted on both FW and contact material that are representative of the grade distribution that will be processed. Based on these tests, the relationship between feed grade and the recovery of metals to concentrate—at the mining company's target concentrate grade—will be confirmed. An ore agreement will then be finalized.

The terms of the ore sale indicative terms are confidential; however, the process follows a standard ore sale arrangement. When material is mined, it will be accumulated into processing lots at site, which will be sampled via a sampling tower prior to transportation the local mining company's Mill. The grade of the lot will determine the recovery of metals in accordance with the relationships confirmed in the testing phase. Once the material has been treated, concentrate containing the agreed-to metal recoveries at Mill's targeted concentrate grade will be designated for smelting. For this stage of processing, there is a set treatment charge per tonne.

The concentrate expected to be produced will then be subject to separate smelter processing terms to determine the net smelter return (NSR). The smelter indicative terms are standard with payables based on a list of metal accountabilities. The metal accountabilities are calculated based on smelter recovery per metal, less a minimum deduction per metal. Smelting charges are based on a treatment charge per tonne and refining charges for each of the accounted metals.

Through this process, payments for Ni, Cu, Co, Pd, Pt, and Au will be realized.

At the time of publication of this report, sample selection for both FW and contact samples is being conducted, with input from the local mining company.

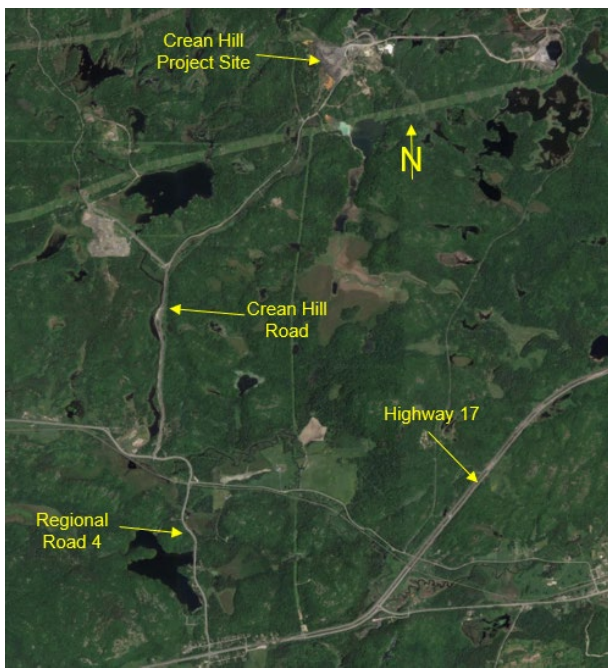
18 PROJECT INFRASTRUCTURE

The Crean Hill Project is in Denison Township within the City of Greater Sudbury, Ontario, Canada, approximately 30 kilometres southwest of downtown Sudbury. The Project will have access to mining-industry leading service providers, suppliers and supply chains, and labour markets available in Sudbury and surrounding communities.

18.1 ROADS

The Project site is accessible year-round by an all-season gravel road from the nearby Regional Road 4 and Trans Canada Highway 17. The all-season gravel road has been the main access to the site for decades and remains suitable for heavy traffic. Materials and equipment will be transported directly to and from the site using typical highway delivery trucks. Workers will use the same roads to commute to and from the site from Sudbury and other nearby communities during project construction and mine operations. The roads leading to the Project site are shown in Figure 18-1.

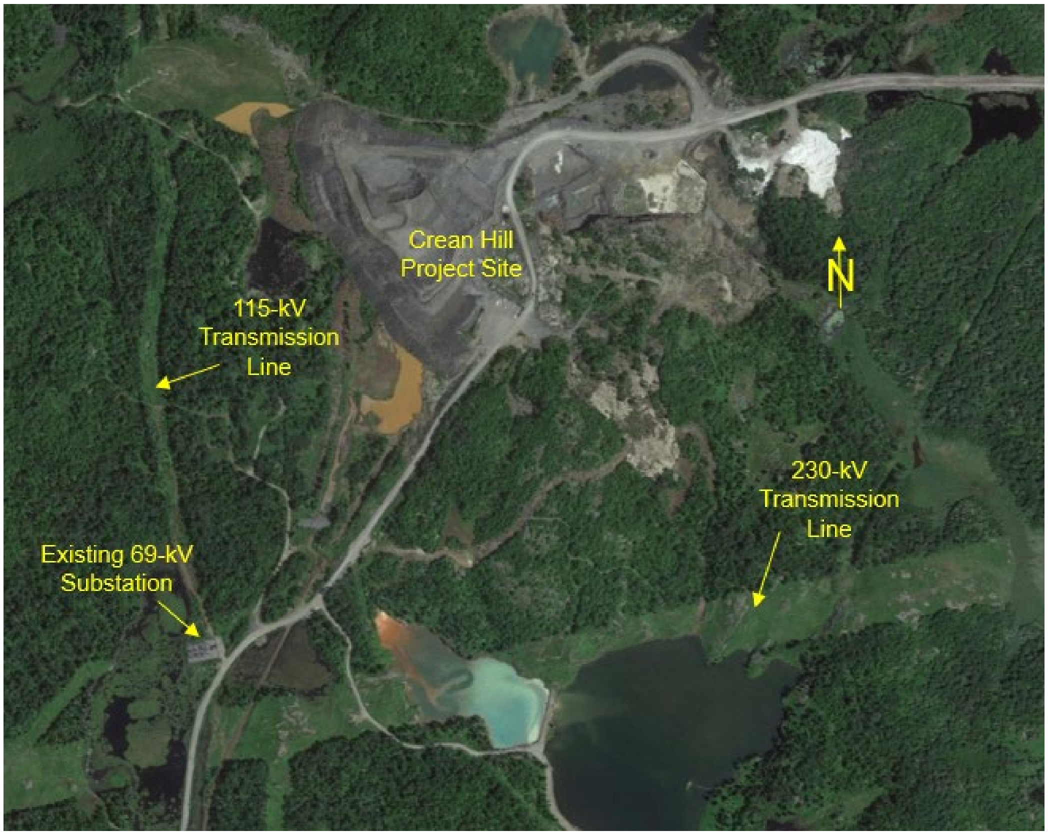
Figure 18-1: Existing Roads to the Crean Hill Project Site



18.2 POWER

A 230-kV power transmission line passes south of the project site and a 115-kV transmission line passes by the western edge of the site. There is an existing substation at the property boundary that may be used to supply power to the site at 69-kV. The adjacent Lockerby mine power infrastructure may also be a potential avenue to supply Crean Hill. An aerial view of the existing power infrastructure near the project site is shown in Figure 18-2.

Figure 18-2: Existing Power Infrastructure Near the Project Site



18.3 WASTE ROCK STOCKPILES

The open pit will mine and haul significant quantities of waste rock to expose the mineralized resource. Stantec has identified two areas on the Crean Hill property that are suitable for waste rock storage on the basis that they are close to the pit and are designed to stay clear of existing waterways and natural ponds. The East and West WRD configuration is shown in Figure 18-4. The East and West WRD have a combined storage capacity of 13.9 million tonnes.

The on-site WRD facilities do not have sufficient storage capacity to store both all waste material from the legacy WRD and the waste mined from the pit. Magna has assumed that all surplus waste (19.7 million tonnes) from the pit can be stored in the Ellen Pit immediately east of the property. Magna will need to confirm the remaining storage capacity of the Ellen Pit and receive permission to store waste. If the pit does not have sufficient capacity, creating an additional waste storage site for excess waste will require additional permitting, site development costs, and possibly increase the cost of waste haulage.

18.4 PROJECT SITE LAYOUT

The initial project site layout will be centered on the facilities required for the ADEX program. The facilities will expand as the project moves through various phases of construction and production. The phases of activity include the ADEX program, project construction period, Phase 1 production from the portal, and

Phase 2 production from the shaft. Where appropriate, the site layout will use existing space that was previously used for similar purposes when the mine was in operation.

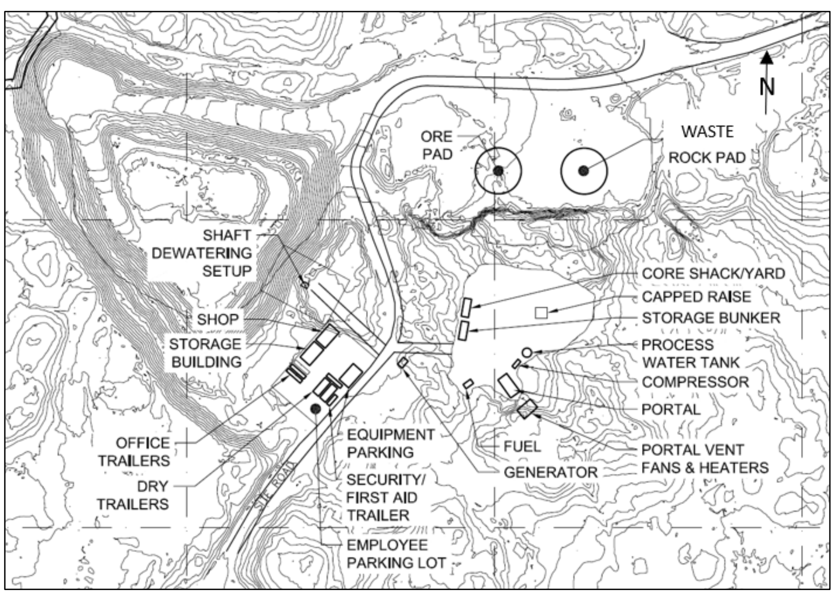
18.4.1 Advanced Exploration Site Layout

The site infrastructure required during the ADEX program is anticipated to include the following.

- Trailers for security/first aid, offices, mine dry, crew assembly, and core shack.
- Maintenance shop.
- Covered storage structure.
- Outside storage bunks.
- Diesel fuel tanks, dispensing, and spill containment area.
- Explosives and detonators storage.
- Mine service water supply pumphouse and storage tank.
- Ventilation fans, mine air heaters, and propane tanks.
- Compressed air plant.
- Diesel generators.
- Storage pads for waste rock and mineralized rock.
- Parking lot for employee personal vehicles.
- Parking area for mobile equipment.

The site layout during the ADEX program is shown in Figure 18-3.

Figure 18-3: Site Layout – ADEX Program



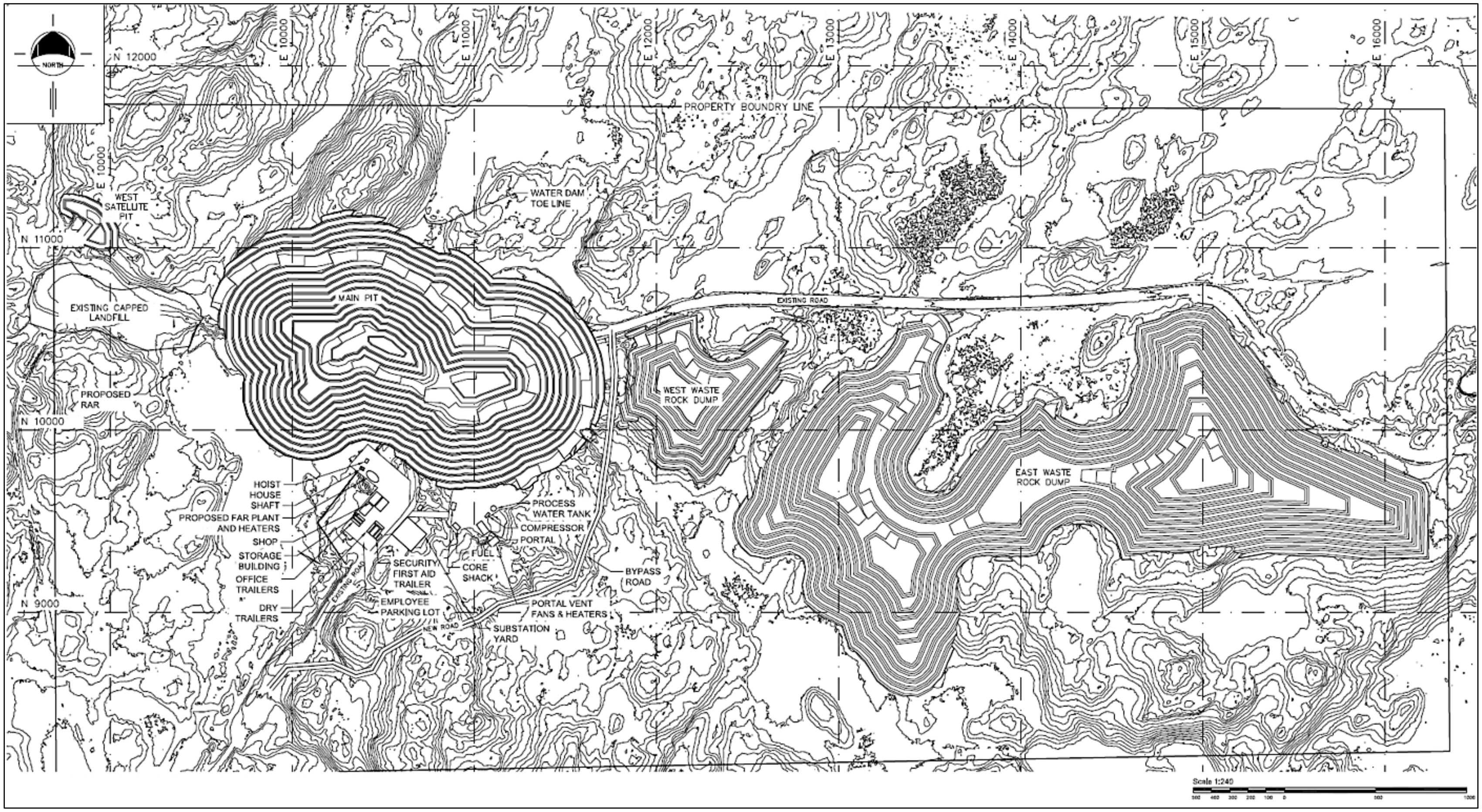
18.4.2 Final Site Layout

The final site infrastructure required during Phase 2 production is anticipated to include the following.

- Trailers for security/first aid/gatehouse.
- Expansion of offices, mine dry, and crew assembly trailer complex.
- Headframe, collar house, and hoist house.
- Crusher and sampling tower.
- Electrical substation and overhead distribution poles.
- Core shack.
- Maintenance shop.
- Covered storage structure.
- Outside storage bunks.
- Diesel fuel tanks, dispensing, and spill containment area.
- Explosives and detonators storage.
- Mine service water and fire water supply pumphouse and storage tank.
- Ventilation fresh air intake fans, mine air heaters, and propane tanks.
- Ventilation return air exhaust fans.
- Slurry plant.
- Compressed air plant.
- Storage pads for waste rock and mineralized rock.
- Parking lot for employee personal vehicles.
- Parking area for mobile equipment.
- New open pit footprint and waste rock dumps.

The final site layout during Phase 2 production is shown in Figure 18-4

Figure 18-4: Site Layout – Phase 2 Production



19 MARKETS AND CONTRACTS

19.1 MARKETS

No project-specific marketing studies were undertaken for the PEA. The PEA discusses the sale of ore from the Crean Hill Project to a third-party mill within trucking distance in the surrounding area. There are two nickel processing facilities (mills) in Sudbury, and one in Timmins, all of which are currently in operation.

The existing mills in Sudbury are located less than 60 km away from the Crean Hill Project. As of April 2023, both Sudbury mills are believed to have capacity to process additional ore. Mineralized material from the Crean Hill Project would be sold to one of these third-party mills and processed into nickel and copper concentrates. These concentrates would be interchangeable commodities for an existing global market and would typically then be processed by a smelter.

There are two smelters in Sudbury which, due to proximity, would likely serve as the destination for processed concentrate. It is assumed that in accordance with the start of mining operations from Crean Hill, a contract would be in place to sell ore to one of the Sudbury-based processing facilities. For purposes of this PEA, no processing charges or penalties at the smelter for deleterious elements and potential out-of-spec concentrate have been assumed and if required, will be included in a negotiated contract with the toll milling facility and smelter.

19.2 CONTRACTS

Vale retains a 3% Net Smelter Return (NSR) royalty from the sale or other distribution of any metals, non-metallic minerals, or other materials mined, produced, or otherwise recovered from the Revised Denison Property (or any waste rock or tailings derived from the Revised Denison Property). Such royalty is to be on, in accordance with, and subject to the terms set out in the Royalty Agreement.

20 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

The Crean Hill Project is located is at the west end of the historic Sudbury mining camp, which has hosted mining and processing for over a century. The Crean Hill Mine on the property operated from 1906 to 2002. Magna acquired partial mining rights to the property in 2022 and has undertaken early exploration activities and economic evaluation work.

20.1 ENVIRONMENTAL STUDIES

The climate in the property area is characterized by moderately long, cold winters and shorter, warm summers, as is typical of continental conditions. The area experiences a wide variation in temperature throughout the year. In winter months, the temperature may drop below -20°C for extended periods. In the summer, the maximum daily temperature may reach over 25°C for extended periods. The daily mean temperatures typically fall below freezing from December through March. Precipitation in the region is characterized as moderate and is generally distributed evenly throughout the year, with only minor seasonal trends. However, the wettest months generally occur from May to October.

Canadian Climate Normals (1981 to 2010) for the Sudbury airport estimate average annual total precipitation at 903 mm, with 676 mm falling as rain and 228 mm (water equivalent) falling as snow. The Sudbury airport is located approximately 45 km northeast of the property. Lake evaporation is estimated at 536 mm using data collected from Rawson Lake (Ontario), located approximately 105 km northeast of the property.

The topography at the property is rugged, with rock knobs representing the dominant bedrock landform in the area. These knobs are often bare or covered with a metre or less of boulder-strewn sandy till, thickening between the highs to between approximately 3 m and 5 m. Slopes are generally steep and complex, and relief ranges between 15 m and 60 m. The exposed rock knobs themselves are well drained. Organic deposits that are often found confined between outcrops are generally observed to be low lying and wet. Drainage to the Vermillion River and several smaller creeks is poor—as a result, small swamps and marshes are numerous.

The property is contained entirely within the Vermillion River watershed. The Vermillion River eventually discharges to the Spanish River, which drains to the North Channel of Lake Huron. Most of the property drains to three separate sub-watersheds that flow into the Vermillion River.

- **Fairbanks Creek Watershed:** The basin forming this watershed is situated to the west and northwest of the property and includes the lakes and tributaries draining to Fairbanks Creek. Major lakes situated in this watershed include Fairbank Lake, Little Fairbank Lake, Skill Lake, and Ethel Lake. Fairbanks Creek flows southward from Fairbank Lake towards the property. This catchment only receives surface water runoff from the western edge of the property, including the west edge of the rehabilitated Crean Hill Mine landfill and a portion of the access road.
- **Northeast Watershed:** This sub-watershed covers the east end of the property. Drainage flows eastward through a series of wetlands and beaver ponds to the Vermillion River. This sub-

watershed receives surface and groundwater discharges from the eastern portion of the property. Diversion dams were constructed to the north and west of the Ellen No. 1 Pit to redirect freshwater to nearby wetlands, which drain to the Vermillion River.

- **Monk Lake Watershed:** The former Crean Hill Mine site is situated in the headwaters of this watershed. Monk Lake is used as a water treatment facility to treat runoff and dewatering from mining activities at the property. A diversion dam was constructed at the north end of this sub-watershed to redirect freshwater northward and ultimately east to the Vermillion River.

As presented in the Crean Hill Mine Closure Plan (Vale, 2022), surface water quality has shown a wide range of concentrations for dissolved metals, as well as sulphate and pH values, demonstrating effects from natural mineralization as well as historic mining activities. Monitoring, reporting, and implementation of mitigation measures in accordance with regulatory requirements is ongoing at the property.

Groundwater flow systems have been identified and characterized around the property, as described in the Crean Hill Mine Closure Plan (Vale, 2022). Groundwater flows away from the site towards the Vermillion River, which is located approximately 2 km to the southeast. Estimated groundwater flow velocities in the overburden flow systems were estimated at between approximately 0.4 to 6 m/yr.

Groundwater quality in overburden has shown a wide range of concentrations for dissolved metals, as well as sulphate and pH values, demonstrating effects from natural mineralization as well as historic mining activities. Groundwater quality for the parameters of interest for the bedrock flow system at the property has either remained the same or has improved over the 2011 to 2018 monitoring period, as documented in the Crean Hill Mine Closure Plan (Vale, 2022). Monitoring, reporting, and implementation of mitigation measures in accordance with regulatory requirements is ongoing at the property.

Three main soil types have been identified on the property. The soils, according to maps produced by the Soil Survey of Canada, include Rockland, Monteagle, and Baldwin (Soil Survey of Canada, 1983).

The property straddles the south range of the Sudbury Igneous Complex (SIC) and hosts part of a large trough structure at the base of the SIC which contains several previously mined deposits, including the ore bodies at Crean Hill Mine, Ellen Mine, and Lockerby Mine, each sitting in embayments (terraces) within the larger trough. Much of the mined Ni-Cu contact mineralization is associated with the embayment structures in the SIC, and the embayments largely control the distribution of nickel-copper mineralization.

Given the abundant mineralization at the property, the mitigation of chemical instability issues (i.e., acid generation, metal leaching) has been the focus of the Water Quality Management Plan for the property that is being implemented by Vale. The Water Quality Management Plan is intended to help Vale make proactive and informed decisions on the management of water quality related environmental risks arising from their mining operations. The management plan establishes a hazard screening and risk evaluation process to assess water quality data for potentially affected water bodies. This process establishes an approach to evaluate and prioritize environmental risks and provides the means to establish remediation priorities and then develop site-specific action plans for mitigation.

Key risk factors used as a guide for prioritization of remediation priorities are as follows.

- Physical setting (surface water and groundwater flow conditions)

- Potential for biological impacts
- Potential for social impacts
- Preliminary assessment of the feasibility of mitigation.

The evaluation of the potential for biological impacts considers metal concentrations in water, the bioavailability of the metals, the presence of substances known to reduce toxicity including the underlying geochemistry of the receiver (e.g., levels of calcium, magnesium and dissolved organic carbon), biotic factors such as acclimation and adaptation, physical setting, and size and location of the receiver. The evaluation of social impacts includes factors such as visibility, potential for health risks, impact on use or enjoyment of the water resource, proximity to urban areas, and proximity to areas of importance to Indigenous communities.

Aquatic resource inventory work has been on-going at the property in accordance with regulatory requirements including the *Metal and Diamond Mining Effluent Regulations* (MDMER). Environment and Climate Change Canada's Environmental Effects Monitoring (EEM) guidance documents have been followed for recent studies.

- Despite concentrations of various metals in sediment being elevated above recognized thresholds due to naturally occurring mineralization and historic mining activities, the first two EEM studies found no effects within the benthic invertebrate communities from effluent discharge. However, the lack of confirmation in the benthic invertebrate community responses between the EEM studies warranted further work to meet MDMER requirements. This work was completed and submitted to Environment and Climate Change Canada in 2019 for review.
- As part of ongoing EEM work, fish sampling studies have been undertaken at the property in 2005, 2007, and 2012.

The 2012 EEM study replicated the fish population study using northern redbelly dace and creek chub, as fathead minnow were unable to be caught in sufficient numbers. The fish capture work identified the presence of white suckers in the receiving environment as well as a variety of small-bodied fish (i.e., fathead minnow, pearl dace, brook stickleback, Iowa darter, fine-scale dace, creek chub, northern redbelly dace, brow bullhead, central mudminnow, brassy minnow, and common shiner). The study did not observe any significant effects on creek chub. It did observe a difference greater than the Critical Effect Size on gonad size in northern redbelly dace, which had become significantly smaller since the previous study. Effects on liver size and condition factors were not confirmed in this study. The lack of confirmation in the fish population responses between the EEM studies warranted further work to meet MDMER requirements. This work was completed and submitted to Environment and Climate Change Canada in 2019 for review.

Background flora and fauna studies for the property were not conducted prior to initial development by the Canadian Copper Company in 1905. A site characterization for the property was conducted in 1993 to inventory and sample foliage from the existing vegetation. As documented in the Crean Hill Mine Closure Plan (Vale, 2022), the levels of metals in the vegetation sampled from around the former mining sites are slightly elevated.

Animal life at the property has not been surveyed recently. However, wildlife species such as moose, bear, deer, ruffed grouse, ducks, otter, beaver, and muskrat are common, based on observations by personnel at the site and knowledge from trappers in the area. Presently, Magna is not aware of any species at risk or habitat features at the property that warrant consideration under Ontario's Endangered Species Act.

Culturally sensitive areas and areas with a high potential to host an archaeological or cultural heritage value have not been defined to date. Going forward, Magna will consult the proximal Indigenous communities and review available electronic databases (e.g., Natural Heritage Information Center) to identify any areas of cultural or heritage significance.

20.2 PERMITTING

The environmental assessment (EA) and permitting framework for metal mining in Canada is well established. The EA processes provide a mechanism for reviewing projects to assess potential impacts to the environment. A comprehensive permitting process is then completed to allow operations to proceed. The Project is regulated through all phases (construction, operation, closure, and post-closure) by both federal and provincial agencies.

Vale has been conducting environmental studies at the property in accordance with regulatory requirements for decades and data collection is ongoing. Cumulatively, environmental studies generally cover the following areas of study.

- Surface water quality and hydrology
- Ground water quality and hydrogeology
- Geochemistry
- Terrestrial and aquatic biology

Magna will complete a gap analysis to determine if supplemental studies are warranted to support the permitting process for the development of the Project. Environmental studies characterize the biophysical environment prior to development of the Project. Technical studies to date have not identified biophysical or cultural heritage values that pose a material constraint to development. The remaining environmental liabilities at the property are primarily the chemical stability risks associated with mine rock piles and decant water from the mine workings (open pits and underground).

This section summarizes permitting requirements for the Project.

20.2.1 Federal Impact Assessment Requirements

The maximum production by previous operators is assumed to be around 5,000 metric tonnes per day. The average production presented in the PEA is below the assumed 5,000 tonne per day threshold in Section 18 of the Physical Activities Regulations made under the Impact Assessment Act. Future studies will confirm the threshold value and the mine plan will consider a production rate that does not trigger a federal impact assessment. Other physical activities that can be subject to a federal impact assessment include transportation facilities (Sections 46 to 55) and water projects (Sections 58 to 61), but these components of the Project are below the applicable thresholds.

The Project is not required to complete a federal Impact assessment if the production rate is kept below the required threshold. However, under Section 9(1) of the Impact Assessment Act, the Minister of Environment and Climate Change may designate a physical activity that is not prescribed by the Physical Activities Regulations if, in their opinion, either the carrying out of that physical activity may cause adverse effects within federal jurisdiction or adverse direct or incidental effects, or public concerns related to those effects warrant the designation.

20.2.2 Provincial EA Requirements

The Project will need to complete the class EA processes listed below, which may be amended in the future by the provincial government.

- Class EA for Resource Stewardship and Facility Development, in accordance with MNR (2003), in advance of issuing permits for Crown timber harvesting, water crossings (>5km^2 watershed area), occupying and constructing on Crown owned surface rights, online dams, and aggregate pit development.
- Class EA for any land tenure decisions, administered by Ministry of Mines
- Category B Class EA is required under the *Electricity Projects Regulation* (O. Regulation 116/01) for the use of diesel generators ($\geq$1MW and <5MW cumulative capacity).
- An evaluation will be required for potential EA requirements under the *Electricity Projects Regulation* (O. Regulation 116/01) and possible approvals from the Ontario Energy Board for potential upgrades to electricity and natural gas service.

20.2.3 Permit Requirements

The provincial government permits that are anticipated to be required for the Project are listed in Table 20-1, depending on the final Project details.

The federal government permits that are anticipated to be required for the Project are listed in Table 20-2, depending on the final Project details.

The municipal government permits that are anticipated to be required for the Project are listed in Table 20-3, depending on the final Project details.

Two permits that are currently in place for the property are the following.

- Crean Hill Mine Closure Plan
- Sewage ECA 6763-9ZXQWA for the Crean Hill and Ellen Pit Wastewater Treatment Facility

It is possible to amend these permits to support the Project. Magna will consult with government agencies as planning progresses to confirm permit requirements.

Table 20-1: Concordance Table for Section 20 Requirements

Requirement of 43-101	Comment
Discuss reasonably available information on environmental, permitting, and social or community factors related to the Project.	Provided in Section 20.1
A summary of the results of any environmental studies and a discussion of any known environmental issues that could materially impact the issuer's ability to extract the mineral resources or mineral reserves.	Provided in Section 20.1, 20.3.1
Requirements and plans for waste and tailings disposal, site monitoring, and water management both during operations and post mine closure.	Provided in Section 20.3, 20.5
Project permitting requirements, the status of any permit applications, and any known requirements to post performance or reclamation bonds.	Provided in Section 20.2
A discussion of any potential social or community related requirements and plans for the Project and the status of any negotiations or agreements with local communities.	Provided in Section 20.4
A discussion of mine closure (remediation and reclamation) requirements and costs.	Provided in Section 20.5 Closure costs are not stated in Section 20. It is understood that these costs are included in the economic model for the PEA

Table 20-2: Federal Permits

Permit	Applicable Act	Responsible Agency	Description
Harmful Alteration, Disruption or Destruction of Fish Habitat	*Fisheries Act*	Fisheries and Oceans Canada	Effects of mine dewatering on fish-bearing surface water features under evaluation
Works in Navigable Waters	*Navigation Protection Act* *(formerly Navigable Waters Protection Act)*	Transport Canada	Authorizations for crossings and/or work in navigable waterway
Schedule 2 Listing, Metal Mining Effluent Regulation	*Fisheries Act*	Environment and Climate Change Canada	Overprinting of water frequented by fish by tailings and mine rock stockpiles (or other deleterious material) will require a listing under Schedule 2 of the *Metal and Diamond Mine Effluent Regulations*, pursuant to the *Fisheries Act*
Manufacturing, storage, and transportation of explosives	*Explosives Act*	Natural Resources Canada	On-site explosives production facility and on-site product storage. Required setbacks are defined in National Standard of Canada Explosives Quantity – Distances CAN/BNQ 2910-510/2015
Migratory Birds	*Migratory Birds Convention Act*	Environment and Climate Change Canada	Prohibition, harm, or disturbance to migratory birds
Transportation of Dangerous Goods	*Transportation of Dangerous Goods Act*	Transport Canada	There are no unique dangerous goods used at the Project. The Project also uses conventional fuels (diesel, gasoline, natural gas, propane) and commercially available welding gases
Species at Risk	*Species at Risk Act*	Environment and Climate Change Canada	Harm, or disturbance to species as designated under the *Species at Risk Act*
Radioisotope License	Nuclear Safety Control Act	Canadian Nuclear Safety Commission	Authorization for nuclear density gauges / X-ray analyzer

Note: These are the relevant permit requirements anticipated based on current understanding of the project presented in the PEA. Further discussion with agencies will be necessary once the project is further defined.

Table 20-3: Municipal Permits

Permit	Applicable Act	Responsible Agency	Description
Building Permits	*Ontario Building Code*	City of Greater Sudbury	Required for applicable structures
Zoning Designation / Re-Zoning	*Planning Act* and Municipal By-Laws (to be determined)		Potential requirement if the zoning designation of the selected site is not compatible with an industrial land use
Road User's Agreement	*Planning Act* and Municipal By-Laws (to be determined)		Agreement to establish cost sharing for road upgrades and maintenance for life of Project

Note: These are the relevant permit requirements anticipated based on current understanding of the project presented in the PEA. Further discussion with agencies will be necessary once the project is further defined.

20.3 ENVIRONMENTAL ASPECTS AND SENSITIVITIES

The project is in the traditional territory of Whitefish Lake First Nation, Sagamok First Nation, and Atikameksheng Anishinawbek First Nation. Magna will seek guidance from the Crown regarding the potential interests of other Indigenous communities. The property is a brown field site and Magna will not be creating new impacts on undisturbed land areas. Emphasis has been placed on reducing water taking, water discharge, fugitive dust, and noise to minimize the potential for off-site impacts that could affect traditional uses and treaty rights.

Management plans that will be developed for the project are as follows.

- Emergency Response Plan
- Spill Prevention and Contingency Plan
- Hazardous Substances Management Plan
- Waste Management Plan
- Construction Environmental Protection Plan
- Air Quality Management Plan
- Water Management Plan
- Environmental Monitoring Plan
- Fish Habitat Compensation Agreement (possibly)
- Adaptive Management Plan
- Fugitive Dust Best Management Plan
- GHG Management Plan
- Engagement Management Plan
- Soils Management Plan
- Waste Rock Management Plan

Environmental aspects that are the focus of current planning efforts are described in the following sections.

20.3.1 Mine Rock Geochemistry

Potentially acid generating and/or metal leaching waste rock will be identified and segregated using lithology and sulphur content, so that it can be managed in accordance with an appropriate management plan. Current information regarding the risk of acid rock drainage / metal leaching (ARD/ML) from mine wastes at Crean Hill was based on historical sampling of waste rock from the former Waste Rock Dump located southeast of the Main Pit (Senes, 2004). One hundred and sixty-two waste rock samples were submitted for limited geochemistry test work. Sixty percent of these samples (97/162) would be classified as at potential risk of acid generation based on current regulatory guidance (MEND, 2009). Total sulphur concentrations were less than 1% wt. Acid neutralising capacity was also low.

A geochemistry program to further evaluate the ARD/ML risk from mine wastes at Crean Hill is currently underway with initial results expected to be available later in 2023.

20.3.2 Water Discharge

Water discharge will be minimized by recycling water to the extent practical using industry standard practices. Surplus water that is not needed for the mining process will be treated using the existing sewage works at Crean Hill Mine in accordance with the issued ECA (as may be amended).

Operational strategies that will be employed to help meet effluent criteria include chemical conditioning in the mine to reduce suspended solids, as well as good blasting practices and use of explosive emulsions in the mine to minimize ammonia with aeration, and biological oxidation of ammonia to prevent elevated nitrogen compound concentrations and solubilization of phosphorus from sediment under anoxic conditions.

20.3.3 Fugitive Dust

Air emission sources will comprise diesel-fueled equipment, diesel generators, combustion heating units, as well as fugitive dust emissions from vehicle operation, crushing, and on-surface material handling typically associated with a mining and crushing 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 tree lines as windbreaks to the maximum extent practical.
- Enclose material transfer points to prevent exposure to wind and use water sprays to suppress dust.
- Other applicable best practices listed in Environment Canada (2009).

Magna will develop and implement a management 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 20-1. The ESDM report demonstrates compliance with MECP air quality criteria during worst-case scenarios.

20.3.4 Site Plan Layout

The open pit is proposed to be in the west-central area of the site with a portion within the Fairbanks Creek watershed and the balance within the Monk Lake watershed. The open pit avoids watercourses and waterbodies and is downgradient from a diversion dam constructed to divert south draining runoff to Fairbanks Creek. The open pit also avoids the historic closed Crean Hill Mine landfill.

The waste rock storage area is proposed to be within a portion of the Northeast watershed draining to the Vermillion River. The waste rock stockpile has been sited as two piles to provide a setback environmental buffer from a watercourse draining southeast. The fish-bearing status of this watercourse is uncertain and will require confirmation in future stages. However, the waste rock piles avoid interference to address concerns with a *Fisheries Act* Schedule 2 prohibition regarding the deposition of mine waste in waterbodies frequented by fish. Ore will be processed, and tailings disposed of at existing processing and tailings storage facilities offsite.

20.4 SOCIAL AND COMMUNITY

The property is in the Sudbury Forest Management Unit and hosts wilderness, forestry, and mineral development land uses.

The site has been developed and is currently managed as it is in active closure. However, there continues to be access through the site for Vale's management of waste rock and Ontario Hydro access to powerline infrastructure. The site is fenced, and natural features are used for safety and to reduce public access. It is not an active hunting or trapping area.

Nevertheless, Magna has engaged with the potentially impacted communities to provide them with an update on our proposed activities. Once the PEA is complete and the required permits identified, Magna will again engage with those communities identified by the Crown and those who have expressed interests in the project area.

One or more public information sessions will be held to promote the opportunities that the Crean Hill mine project will bring to the area, along with participation in regional mining and local business conferences.

20.4.1 Indigenous Consultation

Aboriginal and treaty rights of Indigenous communities are protected under Section 35 of Canada's *Constitution Act.* The federal and provincial governments share the duty to consult Indigenous communities regarding developments such as this Project as part of the environmental assessment and approvals process. Magna has initiated early communications and consulted with proximal First Nations.

The Ministry of Mines will provide guidance to Magna regarding the consultation that is required for the Project and the aspects of the consultation process that will be delegated to Magna. Magna will then prepare an Indigenous Consultation Work Plan in accordance with the requirements of the *Mining Act*, while endeavouring to meet the consultation requirements of the other involved government agencies.

20.4.2 Public Consultation

Consultation with the local and regional communities has commenced and will continue as the Project progresses. This will include meetings 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 Magna's proposed activities, comment and input can be received, and concerns can be resolved in an efficient manner.

20.5 CLOSURE

For the Project to proceed to development, the existing closure plan will be amended to meet requirements of Ontario Regulation 240/00 (refer to Table 20-1) and are consistent with any traditional land uses and occupancy by Indigenous communities.

A closure plan outlines how the project lands will be rehabilitated to a physically and chemically stable, productive land use post closure. The closure plan will meet the requirements of the *Mine Rehabilitation Code of Ontario* (Code) and describe the costs associated with doing so, as well as implementing a monitoring program. Closure plans must be amended periodically during the life of a mine if material changes are made.

To see to it 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 Ministry of Mines. Financial assurance must be included with the submission of a closure plan.

Further to Section 20.1.4, based on precedent, it will be possible to amend the Crean Hill Mine Closure Plan for the Project to proceed. Alternatively, a filed closure plan can be adjusted so that some of the lands become subject to a new closure plan. The path forward will be determined in consultation with the Ministry of Mines.

Elements of the rehabilitation work that would be integrated into the closure plan / closure plan amendment are summarized as follows.

- Buildings, infrastructure, and equipment that are not required for long-term water management will be removed and salvaged, recycled, or disposed of. Contractor owned items and leased items will be removed by the respective owners.
- Contaminated soil will be managed in accordance with MECP requirements.
- The development footprint 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 site and the area that will be re-vegetated using a suitable seed mix prior to planting seedlings consistent with the surrounding plant community. The proportion of rock exposure and vegetation cover will conform to the local landscape.
- Open pits will be flooded following any potential in-pit backfilling to manage mine rock. Boulder fencing will be constructed along any high walls.
- Openings to underground mine workings will be sealed in accordance with the Rehabilitation Code prior to flooding with pH adjusted water.
- Mine rock piles will be built with overall embankment slopes that are adequate for long-term physical stability so that no re-contouring is required at closure. Available stockpiled soil will be used to vegetate waste rock piles as practical to conform to the local landscape. Mine rock types that pose a chemical stability risk will be managed in accordance with the management plan that will be developed as part of the closure plan.
- Final removal of power distribution and water management infrastructure once active water management is no longer required.

Roads will be rehabilitated in accordance with MNR (1995) and removed from use. Roads that are not requested to remain in place by MNRF, a First Nation, or a third party will be removed in accordance with MNR (1995) and any supplemental guidance from MNRF, with financial assurance provided in the closure plan.

21 CAPITAL AND OPERATING COSTS

The cost estimate for the Preliminary Economic Assessment on the Crean Hill Project was prepared to a Class 5 as defined by AACE International, with an approximate -20 to -50% / +30 to +100% accuracy. All costs in the estimate are reported in Quarter Two (Q2) 2023 Canadian dollars.

The costs are separated into four main cost classifications.

- Advanced Exploration (ADEX) classification covers:
  - All direct and indirect costs incurred during the ADEX program. The ADEX program is anticipated to extend for 13 months from Q4 2023 to Q4 2024.
  - Contingency.
  - Capitalized operating costs.
- Project Capital classification covers:
  - All direct and indirect costs after the ADEX program to the project period end date. The project period is anticipated to extend from Q4 2024 to Q3 2025 when substantial completion of open pit and underground mine infrastructure to sustain production at approximately 60% of design capacity is achieved and sustainable.
  - Contingency.
  - Capitalized operating costs.
- Sustaining Capital classification covers:
  - All direct and indirect capital costs after the project period end date to continue producing from the open pit and during underground Phase 1 and expansion into Phase 2 production.
- Operating Costs classification covers:
  - All direct and indirect operating costs incurred after the project period end date.

21.1 CAPITAL COST ESTIMATE

The capital cost estimate has been separated into costs for the underground mine and open pit mine.

21.1.1 Underground Mine ADEX and Capital Costs

The estimated ADEX and capital costs for the underground mine are summarized in Table 21-1.

Table 21-1: Estimated Underground Mine Capital Costs

Item	ADEX ($ millions)	Project Capital ($ millions)	Sustaining Capital ($ millions)	Total ($ millions)
Site Preparation	$0.12	$0.77	$0.18	$1.07
Dewater Flooded Workings	$0.71	$1.05	$1.05	$2.80
Portal Construction	$0.32	$0.00	$0.00	$0.32
ADEX Lateral Development	$12.22	$0.00	$0.00	$12.22
Block 1 Capital Development	$0.00	$6.01	$14.41	$20.42
Block 2 Capital Development	$0.00	$9.00	$23.87	$32.87
Block 3 Capital Development	$0.00	$0.00	$31.05	$31.05
Block 4 Capital Development	$0.00	$0.00	$52.60	$52.60
Raise Development	$0.00	$2.47	$9.78	$12.25
Mobile Equipment - Contractor Rentals	$2.60	$3.00	$0.00	$5.60
Ventilation Plants / Fans / Controls	$0.97	$0.33	$22.04	$23.34
Shaft and Material Handling	$0.00	$1.17	$76.31	$77.48
Dewatering (excludes excavations)	$0.08	$0.44	$1.53	$2.05
Service Water (excludes UG piping)	$0.27	$0.15	$0.00	$0.42
Compressed Air (excludes UG piping)	$0.10	$0.20	$0.00	$0.30
UG Facilities (excludes excavations)	$0.00	$0.65	$4.43	$5.07
Electrical Distribution (excludes excavations)	$1.58	$8.10	$8.73	$18.41
Surface Facilities	$1.20	$1.30	$1.20	$3.70
Total UG Mine Direct Capital Costs	**$20.18**	**$34.63**	**$247.16**	**$301.97**
Engineering, Procurement, Construction Management (EPCM) (5%)	$1.01	$1.73	$0.00	$2.74
Project Indirects (15%)	$3.03	$5.19	$0.00	$8.22
Contingency – ADEX 35%	$8.47	$0.00	$0.00	$8.47
Contingency – Project Capex 35%	$0.00	$14.55	$0.00	$14.55
Total UG Mine Direct and Indirect Capital Costs	**$32.69**	**$56.11**	**$247.16**	**$335.96**
Underground Mine Closure	$0.00	$0.00	$0.00	**$5.85**

The underground mine capital costs were estimated using the methodology outlined in Table 21-2.

Table 21-2: Underground Mine Capital Cost Estimating Methodology

Item	Estimating Methodology
Site Preparation	Surface pad areas measured from site plans and unit costs for grubbing / levelling / filling applied.
Dewatering Flooded Workings	Water volumes estimated from the mine model of historic voids with assumptions for volumes that are backfilled versus flooded. Pumping calculations to select an appropriate pump size/type and pumping rate. Estimated costs to prepare / set up for dewatering, purchase pumps/materials, maintain pumps.
Portal Construction	Assumed proposed outcrop/highwall face is available with allowance for drill/blast/fill to prep the face, install new ground support, and collar the ramp socket.
Lateral and Vertical Development	Development quantities from the Deswik mine model, including design allowances. Advance rates estimated from development cycle times and anticipated efficiencies. Direct cost per foot of development estimated using labour rates, material prices, and equipment operating costs from Stantec's experience and database. Includes contractor mark-ups/margins where appropriate. Excludes contractor equipment rentals.
Mobile Equipment - Contractor Rentals	Monthly rental rates for the contractor equipment with appropriate mark-ups.
Ventilation Plants / Fans / Controls	Ventilation fan selection based on the VentSim model. Fans, Heaters, Propane farm purchase costs and construction costs estimated from similar applicable installations from other studies and projects and discussions / quotes with suppliers.
Shaft & Material Handling	Shaft stripping and re-equipping direct unit cost estimated using labour rates, material prices, and equipment operating costs from Stantec's experience and database. Equipment purchase costs and construction costs estimated from similar applicable installations from other studies and projects. Includes contractor mark-ups/margins.
Dewatering (excludes excavations)	Estimated the service water introduced into the mine and used reported historic pumping rates to assume ground water inflow. Preliminary pump calculations to size pumps. Pump costs and sump/pumproom construction cost estimates from recent similar applicable installations from other studies and projects. Piping included in the development costs and shaft costs. Excavation included in the lateral development quantities.
Service Water (excludes UG piping)	Estimated the service water requirements for each phase of activity. Sized piping for main ramps and on sublevels. Piping included in the development unit rate.
Compressed Air (excludes UG piping)	Estimated the compressed air requirements for each phase of activity. Sized piping for main ramps and on sublevels. Piping included in the development unit rate.

Item	Estimating Methodology
UG Facilities (excludes excavations)	Estimated the facilities required and used costs for typical installations from other Stantec studies and projects. Excavations included in the lateral development quantities.
Electrical Distribution (excludes excavations)	Estimated the power consumption based on annual power users and sized equipment. Equipment purchase costs and construction costs estimated from similar applicable installations from other studies and projects. Typical power and communications cables included in the development unit rate. Large power distribution cables estimated based on mining quantity take offs (QTOs). Excavation included in the lateral development quantities.
Surface Facilities	Estimated the facilities required and used costs for typical installations from other Stantec studies and projects.
Closure Costs	Identified closure items related to the underground mine and estimated costs based on Stantec experience and database.
EPCM	Applied 5% to the capital costs during ADEX and Project Period
Project Indirects	Applied 15% to the capital costs during the ADEX and Project Period
Contingency – ADEX	Applied 35% contingency to the estimated capital costs, EPCM costs, and Project Indirect costs incurred during the ADEX program.
Contingency – Project Capex	Applied 35% contingency to the estimated capital costs, EPCM costs, and Project Indirect costs incurred during the Project Period.
Escalation	No escalation applied.

The direct lateral development rates used in the capital cost estimate include the following cost components:

- Direct labour (three-person crew for drilling, blasting, mucking, ground support installation, and piping services installation).
- Drilling and blasting, which includes:
  - Jumbo carrier and drill operating costs (excluding maintenance labour).
  - Drilling consumables (e.g., bits, steel).
  - Explosives loader operating costs.
  - Blasting consumables (e.g., explosive, detonators).
- Mucking, which includes:
  - LHD operating costs (excluding maintenance labour).
  - Note: Truck haul costs are excluded from the direct development cost.
- Ground support, which includes:
  - Mechanical bolter operating costs (excluding maintenance labour).
  - Shotcrete sprayer operating costs.
  - Drilling consumables (e.g., bits, steel).
  - Ground support consumables (e.g., bolts, plates, screen, shotcrete).
- Services, which includes:
  - Service water, compressed air, and dewatering pipe (where required).
  - Power cable and communications cable.

- Contractor mark-up, which includes:
  - Development completed by the contractor including an 18% markup on labour, materials, and equipment operating costs.

All indirect costs such as power consumption, equipment maintenance labour, supervision, personnel carriers, and technical services are included in the operating costs.

The direct capital development unit rates are summarized in Table 21-3.

Table 21-3: Direct Capital Lateral Development Unit Rates

Heading Type	$/foot
Ramp – Single Face – Contractor Learning Curve	$2,232
Ramp – Single Face – Contractor	$1,804
Ramp – Multi Face – Contractor	$1,500
Lateral Waste – Multi Face – Contractor	$1,255
Rehab Existing Drifts – Contractor	$879
Lateral Waste – Multi Face – Owner	$886
Rehab Existing Drifts – Owner	$620

21.1.2 Open Pit Mine Capital Costs

The estimated capital costs for the open pit mine are summarized in Table 21-4.

Table 21-4: Estimated Open Pit Mine Capital Costs

Item	ADEX ($ millions)	Project Capital ($ millions)	Sustaining Capital ($ millions)	Total ($ millions)
Move existing Waste Stockpile	$0.00	$14.15	$0.00	$14.15
Clear and Grub Open Pit Forested Area	$0.00	$0.25	$0.00	$0.25
Clear and Grub Waste Stockpile Area	$0.00	$0.81	$0.90	$1.71
Water Collection Ditching	$0.00	$0.22	$0.17	$0.39
Open Pit Bypass Road	$0.00	$0.00	$0.63	$0.63
Total Open Pit Mine Direct Capital Costs	$0.00	**$15.42**	**$1.70**	**$17.13**
EPCM (5%)	$0.00	$0.77	$0.00	$0.77
Project Indirects (15%)	$0.00	$2.31	$0.00	$2.31
Contingency – ADEX (35%)	$0.00	$0.00	$0.00	$0.00
Contingency – Project Capex (35%)	$0.00	$6.48	$0.00	$6.48
Total Open Pit Mine Direct + Indirect Capital Costs	**$0.00**	**$24.98**	**$1.70**	**$26.69**
Closure Costs (Open Pit Related)	$0.00	$0.00	$0.00	**$11.24**

21.2 OPERATING COSTS

The operating cost estimate for the underground mine and open pit mine have been reported separately.

21.2.1 UG Mine Operating Costs

The estimated underground mine operating costs are $119.82 per metric tonne mined and are summarized in Table 21-5.

Table 21-5: Estimated Underground Mine Operating Costs

Item	ADEX Capitalized Opex ($ millions)	Project Capitalized Opex ($ millions)	Operating Period ($ millions)	Total ($ millions)
Block 1 Operating Development	$0.00	$0.00	$12.99	$12.99
Block 2 Operating Development	$0.00	$1.34	$30.92	$32.26
Block 3 Operating Development	$0.00	$0.00	$22.41	$22.41
Block 4 Operating Development	$0.00	$0.00	$67.16	$67.16
Block 1 Stoping	$0.00	$0.00	$15.11	$15.11
Block 2 Stoping	$0.00	$0.27	$47.00	$47.26
Block 3 Stoping	$0.00	$0.00	$22.39	$22.39
Block 4 Stoping	$0.00	$0.00	$129.47	$129.47
Production Truck Haulage	$0.04	$0.24	$27.31	$27.59
Waste Rock Truck Haulage	$0.54	$1.50	$9.93	$11.97
Backfill	$0.00	$0.21	$86.88	$87.09
Mobile Equipment – Lease	$0.03	$0.02	$160.94	$160.98
Indirect Equipment Operating – Owner	$0.02	$0.01	$18.55	$18.58
Indirect Labour – Owner	$1.35	$1.59	$234.19	$237.12
Utilities	$11.07	$3.75	$153.54	$168.37
Fixed Plant Maintenance	$0.22	$0.34	$41.61	$42.17
Diamond Drilling	$1.06	$1.00	$15.88	$17.94
Surface Waste Rock Handling	$0.20	$0.27	$0.61	$1.08
Processing	$0.93	$2.44	$783.81	$787.17
Site General and Administrative	$0.22	$0.34	$40.34	$40.90
Total UG Operating Cost	**$15.68**	**$13.33**	**$1,921.02**	**$1,950.03**
Total UG Operating Cost per Tonne				**$119.82/tonne**

The underground mine operating costs were estimated using the methodology outlined in Table 21-6.

Table 21-6: Underground Mine Operating Cost Estimating Methodology

Item	Estimating Methodology
Lateral Development	Stope crosscuts and stope sills and cut and fill attack ramps. Development quantities from the Deswik mine model, including design allowances. Advance rates estimated from development cycle times and anticipated efficiencies. Direct cost per foot of development estimated using labour rates, material prices, and equipment operating costs from Stantec's experience and database. Includes contractor mark-ups/margins where appropriate.
Longhole Stoping	Stoping quantities from the Deswik mine model, including dilution and mining recovery. Production rate estimated from stoping cycle time and anticipated efficiencies. Direct unit costs estimated using labour rates, material prices, and equipment operating costs from Stantec's experience and database.
Cut and Fill Stoping	Used same approach as development direct costs.
Development Truck Haulage	Development tons from the Deswik mine model. Estimated haul cycle times and productivities from various sublevels to surface and to other sublevels for backfill. Estimated unit cost based on productivities, labour rates, and equipment operating costs from Stantec's experience and database.
Production Truck Haulage	Stope tons from the Deswik mine model for ADEX, Block 1, and Block 2. Lateral transfer tons from passes to shaft on 4000 L for Block 3 and Block 4. Estimated haul cycle times and productivities from various sublevels to surface and for the lateral transfer on 4000 L. Estimated cost per ton hauled based on productivities, labour rates, and equipment operating costs from Stantec's experience and database.
Backfill	Backfill quantities from Deswik model. Weighted average of cemented rockfill and unconsolidated rockfill to estimate a single unit rate per ton of fill placed. No backfill in Uppers stopes. Direct costs per ton estimated using labour rates, material prices, and equipment operating costs from Stantec's experience and database.
Mobile Equipment Leases	Recent budget purchase quotes from other Stantec projects assumed leased over 48 months at 8% annual percentage rate (APR). Direct mobile equipment fleet units (jumbo, bolter, LHD, trucks, drills) determined from the mine plan. Indirect mobile equipment (e.g., personnel carriers, grader, boom truck, scissor lift) estimated based on the scale of the operation.
Indirect Labour	Generated a list of Indirect labour personnel with all-in annual cost applied to each person (e.g., wages, burdens).
Utilities	Estimated diesel generated power consumption during ADEX and start of Project Period and applied a unit rate. Estimated grid power consumption for remainder of the Project Period and operating period and applied a unit rate. Estimated mine air heating propane consumption and applied a unit rate.

Item	Estimating Methodology
Fixed Plant Maintenance	Allowance based on total tons generated in the mine plan (development and production).
Definition Diamond Drilling	Allowance based on production tons.
Surface Waste Rock Handling	Estimated waste rock hauled to surface considering waste that remains UG for backfill. Allowance per ton to move waste rock from portal pad to stockpile and/or return backhaul underground for backfill.
Processing	Production tons from the Deswik mine model. Unit costs to rehandle and crush/sample on site. Surface truck haul to processing plant. Processing cost (based on third party custom toll milling).
Site General and Administrative (G&A)	Non-mining G&A allowance based on total tons generated in the mine plan (development and production). Carbon tax allowance.

The key operating cost unit rates used to estimate the operating costs are summarized in Table 21-7.

Table 21-7: Key Underground Mine Operating Unit Rates

Item	Units	Unit Rate
Stope Cross-cut / Sill Development – Contractor Stope Cross-cut / Sill Development – Owner Cut and Fill Attack Ramp – Owner	$/foot	$1,255 $886 $666
Longhole Stoping – Owner	$/LH stope ton	$6.75
Cut and Fill Stoping – Owner	$/CAF stope ton	$26.41
Production Truck Haul – Block 1 – Owner Production Truck Haul – Block 2 – Owner Production Truck Haul – 4000 L Transfer – Owner	$/ton hauled	$2.77 $2.86 $0.80
Waste Rock Truck Haul – ADEX - Contractor Waste Rock Truck Haul – Block 1 – Contractor Waste Rock Truck Haul – Block 2 – Contractor Waste Rock Truck Haul – Block 1 – Owner Waste Rock Truck Haul – Block 2 – Owner Waste Haul to Stope (backfill) – Owner	$/ton hauled	$2.56 $5.64 $5.26 $4.78 $4.46 $1.40
Backfill (CRF and rockfill weighted average)	$/ton fill	$10.75
Power – Diesel Generated Power – Grid	$/kWh	$0.800 $0.103
Propane	$/litre	$1.00
Fixed Plant Maintenance – Phase 1 Fixed Plant Maintenance – Phase 2	$/total tons	$1.00 $2.00
Definition Diamond Drilling	$/production ton	$1.00
Surface Waste Rock Handling	$/waste ton	$1.00
Rehandle and Primary Crush on Site Secondary Crush and Sample on Site Surface Haul to Processing Plant Custom Toll Milling	$/production ton	$2.00 $1.50 $5.00 $35.38
G&A	$/total tons	$1.00

21.2.2 Open Pit Mine Operating Costs

A mining contractor will provide the labour, equipment, and materials to operate the open pit mine. The estimated open pit mine operating costs are $102.73 per tonne mined and are summarized in Table 21-8.

Table 21-8: Estimated Open Pit Operating Cost Summary

Item	Project Capitalized Opex ($ millions)	Operating Period ($ millions)	Total ($ millions)
Open Pit Waste Rock	$15.90	$168.18	$184.08
Open Pit Production	$0.65	$23.36	$24.01
Processing	$5.02	$180.16	$185.18
Total Open Pit Operating Cost	**$21.57**	**$371.70**	**$393.27**
Total Open Pit Operating Cost per tonne			**$102.73/tonne**

The open pit operating costs have been based on Stantec's study experience with a similar contractor-mined open pit. The key operating cost unit rates used to estimate the operating costs are summarized in Table 21-9.

Table 21-9: Key Open Pit Operating Unit Rates

Item	Units	Unit Rate
Open Pit Waste Rock Mining	$/waste ton	$5.69
Open Pit Production Mining	$/production ton	$5.69
Rehandle and Primary Crush on Site Secondary Crush and Sample on Site Surface Haul to Processing Plant Custom Toll Milling	$/production ton	$2.00 $1.50 $5.00 $35.38

22 ECONOMIC ANALYSIS

A cash flow model was developed by Stantec to assess the potential economic viability of the Crean Hill Project. The cash flow forecasts are based on the mine development and production schedule, estimated capital expenditures, and the estimated operating costs over the life of the project. The revenues and costs are in constant Q2 2023 Canadian dollars (CAD) without escalation.

The cash flow model starts with the site preparation and construction for the ADEX program, excluding permitting and engineering studies, and ends with closure costs. Costs incurred before this time are excluded from the evaluation.

The metal prices and exchange rate used in the cash flow model are shown in Table 22-1.

Table 22-1: Metal Prices and Exchange Rate

Item	Value
Nickel Price	US$9.50 per pound
Copper Price	US$3.50 per pound
Cobalt Price	US$22.00 per pound
Platinum Price	US$1,000.00 per oz
Palladium Price	US$1,800.00 per oz
Gold Price	US$1,700.00 per oz
Exchange Rate	1.30 CAD/US$

22.1 METAL REVENUE

Stantec estimated an NSR value for every tonne of mineralized material. The NSR value is the combined net value of all metals (Ni, Cu, Co, Pt, Pd, and Au) and considers the processing recoveries of the respective metals, treatment, and refining charges; concentrate freight cost; and the payable quantities of metal based on an assumed third-party toll milling arrangement.

The revenue stream in the cash flow forecast is derived from the NSR value per tonne of mined mineralized resource delivered to the mill.

22.2 ROYALTIES

There is a 3% NSR royalty on the project.

22.3 TAXES

Canadian federal and Ontario provincial income tax rates of 15% and 10%, respectively, were applied to the taxable income each year to arrive at the annual income tax paid.

Canadian federal income taxes were calculated using the operating cash flow for the combined underground and Open Pit mines. The Capital Cost Allowance (CCA) and the Canadian Development

Expense (CDE) were deducted from the operating cash flow before tax to arrive at taxable income. The Accelerated Investment Provisions from November 2018 were included in the available CCA and CDE deductions for applicable years. The refundable Investment Tax Credit for Clean Technology Manufacturing was also included in the tax calculations.

Ontario income taxes were calculated using the operating cash flow for the combined underground and Open Pit mine. The annual profit exemption of $500,000 has been applied in calculating the Ontario tax payable.

22.4 CASH FLOW

The Project is a combined underground and Open Pit mining operation. To reflect the impact of each mine on the project economics, a separate cash flow model was prepared for the underground mine, the Open Pit mine, and the combined underground and Open Pit mine. The estimated post-tax net present value (NPV)—discounted 8%—for the Project is $230.4 million, with a 23.37% IRR. The cash flow, NPV, and IRR results are summarized in Table 22-2.

Table 22-2: Cash Flow Results Summary

Item	Underground Mine	Open Pit Mine	Combined Underground and Open Pit
Metric Tonnes Mined/Milled	16,274,220	3,828,385	20,102,605
Average NSR ($/tonne)	$190.36	$131.05	$179.07
Revenue (millions)	$3,098,00	$501.71	$3,599.70
Royalty (millions)	$90.42	$14.42	$104.84
Capital Cost – Includes ADEX (millions)	$335.96	$26.69	$362.64
Operating Cost – includes ADEX (millions)	$1,950,03	$393.27	$2,343,30
Closure Cost (millions)	$5.85	$11.24	$17.09
Pre-Tax Discounted Cash Flow – NPV 8% (millions)	$251.99	$38.44	$290.43
Pre-Tax IRR	22.43%	37.82%	23.91%
Taxes (millions)	$146.86	$16.69	$163.29
Post-Tax Discounted Cash Flow – NPV 8% (millions)	$203.02	$26.12	$230.44
Post Tax IRR	22.39%	29.62%	23.37%

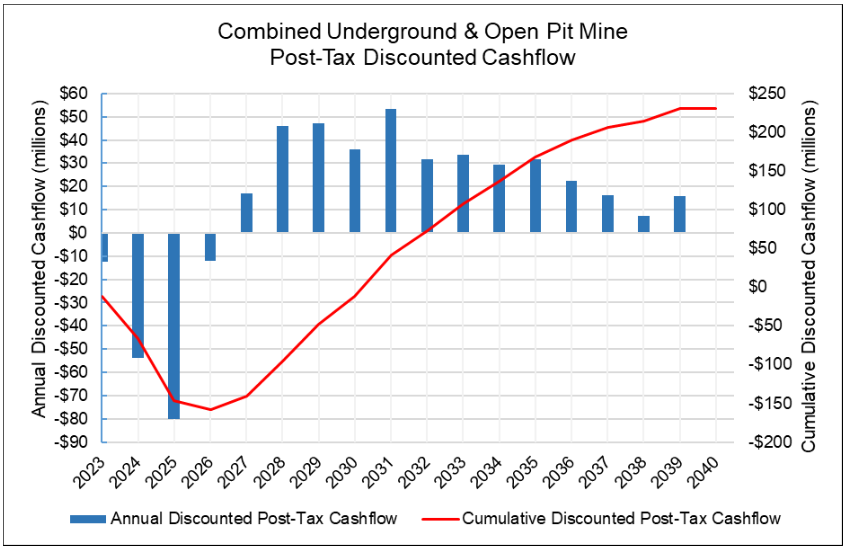
The estimated combined underground and open pit annual mine cash flow is summarized in Table 22-3.

Table 22-3: Combined Underground and Open Pit Mine Estimated Cash Flow

Item	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Tonnes Mined/Milled	**20,102,605**	0	19,273	845,745	1,693,830	1,835,131	2,002,812	1,574,202	1,251,242	1,306,200	1,346,980	1,497,068	1,376,157	1,400,306	1,221,432	964,877	717,899	1,049,452	0
Ni %	**0.624**	0.000	0.676	0.555	0.547	0.550	0.512	0.625	0.656	0.792	0.663	0.648	0.661	0.681	0.666	0.650	0.574	0.655	0.000
Cu %	**0.549**	0.000	0.709	0.462	0.446	0.468	0.461	0.546	0.598	0.751	0.550	0.571	0.562	0.590	0.617	0.605	0.519	0.581	0.000
Co %	**0.022**	0.000	0.020	0.020	0.020	0.020	0.019	0.022	0.023	0.027	0.024	0.023	0.023	0.023	0.022	0.023	0.021	0.023	0.000
Pt (ppm)	**0.569**	0.000	1.455	0.423	0.548	0.441	0.749	0.756	0.661	0.617	0.564	0.493	0.529	0.551	0.473	0.579	0.457	0.496	0.000
Pd (ppm)	**0.621**	0.000	0.713	0.292	0.517	0.490	0.712	0.788	0.798	0.760	0.682	0.577	0.635	0.633	0.546	0.637	0.645	0.485	0.000
Au (ppm)	**0.341**	0.000	0.400	0.173	0.282	0.315	0.374	0.458	0.407	0.413	0.363	0.304	0.320	0.361	0.305	0.332	0.328	0.286	0.000
Average NSR ($/tonne)	**$179.07**	$0.00	$218.92	$141.98	$150.82	$153.09	$155.09	$184.11	$198.00	$231.48	$191.29	$183.28	$187.87	$194.81	$187.54	$191.02	$169.48	$181.07	$0.00
Revenue (millions)	**$3,599.70**	$0.00	$4.22	$120.08	$255.47	$280.94	$310.63	$289.82	$247.74	$302.36	$257.66	$274.39	$258.54	$272.79	$229.06	$184.31	$121.67	$190.02	$0.00
Royalty (millions)	**$104.84**	$0.00	$0.12	$3.46	$7.38	$8.12	$8.99	$8.43	$7.23	$8.85	$7.51	$7.98	$7.53	$7.95	$6.67	$5.37	$3.53	$5.70	$0.00
Capital Cost – Includes ADEX (millions)	**$362.64**	$9.89	$50.86	$81.51	$42.44	$71.68	$41.86	$24.13	$13.10	$8.15	$7.99	$2.66	$3.16	$2.11	$2.77	$0.10	$0.04	$0.20	$0.00
Operating Cost – Includes ADEX (millions)	**$2,343.30**	$2.38	$16.37	$136.53	$227.50	$192.87	$196.30	$171.11	$149.68	$157.31	$159.56	$168.35	$157.94	$156.86	$138.66	$115.40	$87.25	$109.21	$0.00
Closure Cost (millions)	**$17.09**	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$17.09	$0.00
Pre-Tax Discounted Cash Flow - NPV 8% (millions)	**$290.43**	-$12.27	-$58.46	-$86.96	-$17.35	$6.07	$43.20	$54.29	$45.35	$69.19	$41.32	$44.18	$38.56	$42.04	$29.77	$21.60	$9.72	$20.16	$0.00
Pre-Tax IRR	**23.91%**	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Taxes (millions)	**$163.29**	$0.00	-$5.23	-$7.96	-$6.72	-$14.89	-$4.22	$11.12	$15.84	$29.55	$19.14	$22.50	$21.74	$25.98	$20.11	$15.52	$7.41	$14.56	-$1.17
Post Tax Discounted Cash Flow – NPV 8% (millions)	**$230.44**	-$12.27	-$53.62	-$80.14	-$12.02	$17.02	$46.07	$47.28	$36.11	$53.22	$31.75	$33.76	$29.24	$31.73	$22.38	$16.31	$7.39	$15.91	$0.32
Post Tax IRR	**23.37%**	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–

The post-tax discounted (8%) cash flow for the Project (combined Underground and Open Pit Mine) is shown in Figure 22-1.

Figure 22-1: Discounted Cash Flow – Combined Underground and Open Pit Mine



22.5 PAYBACK PERIOD

The Project will start generating revenue from mineralized resource mined during the ADEX program and Capital Project period and will start being cash positive four years from the start of the ADEX program. The estimated payback period of the initial capital invested during the Project Period is approximately 3.3 years from the start of the Project Period.

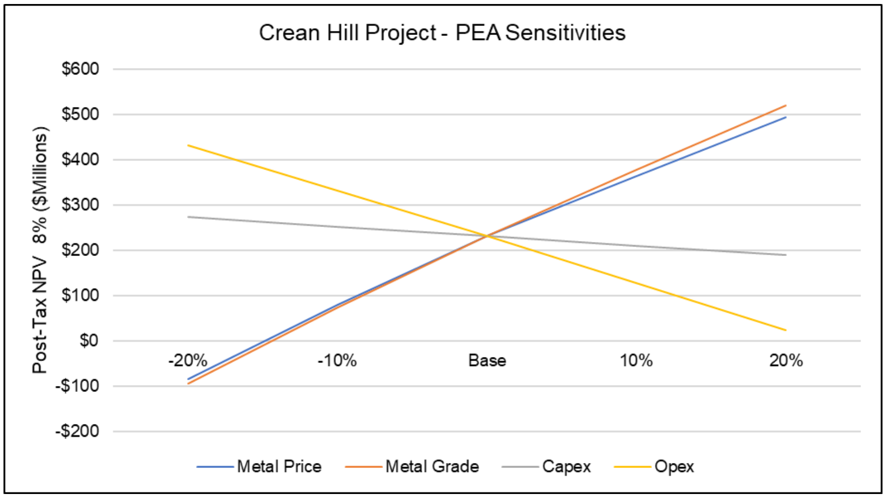
22.6 SENSITIVITY ANALYSIS

The Crean Hill Project is most sensitive to metal price and metal grade, and least sensitive to capital costs, as summarized in Table 22-4 and plotted in the graph in Figure 22-2.

Table 22-4: Crean Hill PEA Sensitivities (Post-Tax NPV 8%, millions)

Variance	Metal Price	Metal Grade	Capital Cost	Operating Cost
-20%	-$85.4	-$94.6	$272.7	$431.4
-10%	$78.9	$73.7	$251.6	$331.6
Base	**$230.4**	**$230.4**	**$230.4**	**$230.4**
10%	$363.0	$376.6	$209.3	$127.6
20%	$492.7	$519.3	$188.2	$22.9

Figure 22-2: Crean Hill PEA Sensitivities Graph



23 ADJACENT PROPERTIES

23.1 ADJACENT PROPERTIES

Properties adjacent to the Crean Hill Project are held by Vale Canada Ltd. and KGHM International. No information on those adjacent properties is necessary to make the technical report understandable and not misleading.

24 OTHER RELEVANT DATA AND INFORMATION

This PEA presents the Crean Hill Project as a combined open pit and underground mine with an assumed ore sale arrangement with a Sudbury-based processing facility, and without consideration of other projects that Magna Mining Inc. (Magna) has in the Sudbury area. The ore sale assumption is considered a conservative approach to demonstrate the Crean Hill Project as a standalone project.

Although not reflected in this PEA, it is Magna's vision that the Crean Hill Project will be part of a Hub and Spoke model with centralized milling at Magna's Shakespeare Project. The Shakespeare Project is currently undeveloped with no milling facilities in place. Processing the Crean Hill Project feed through a Magna-owned facility is anticipated to reduce processing costs while improving the recovery of PGMs. The lower processing cost could result in mining at a lower cut-off value, while the improved recovery could result in a higher NSR.

24.1 SHAKESPEARE HUB AND SPOKE OPPORTUNITY

Based on the open pit and underground mine design work completed for the PEA base case, Stantec prepared an alternative Crean Hill Project scenario considering the Shakespeare Hub and Spoke model to demonstrate the potential opportunity.

The Crean Hill Project's production potential under the Shakespeare Hub and Spoke is summarized in Table 24-1.

Table 24-1: Crean Hill Production Potential with Shakespeare Hub and Spoke Model

Item	Underground	Open Pit	Combined
Tonnes Mined	21,791,858	6,405,636	28,197,495
Average NSR $/tonne	$181.42	$110.00	$165.20
%Ni	0.604	0.431	0.565
%Cu	0.538	0.357	0.497
%Co	0.021	0.016	0.020
Pt (grams per tonne)	0.561	0.348	0.513
Pd (grams per tonne)	0.646	0.215	0.548
Au (grams per tonne)	0.345	0.157	0.302

The Crean Hill Project's financial potential under the Shakespeare Hub and Spoke model is summarized in Table 24-2.

Table 24-2: Crean Hill – Financial Potential with Shakespeare Hub and Spoke Opportunity

Item	Underground (CAD millions)	Open Pit (CAD millions)	Combined (CAD millions)
Project Period Direct Capital Costs	$34.63	$15.53	$50.16
Project Period Indirect Costs	$6.93	$3.11	$10.03
Project Period Contingency	$14.55	$6.52	$21.07
Sustaining Capital Costs	$252.91	$3.80	$256.71
Total Capital Costs	**$309.01**	**$28.96**	**$337.97**
Operating Costs during Project Period	$12.15	$19.70	$31.85
Operating Cost during Operating Period	$2,028.32	$415.24	$2,443.56
Total Operating Costs	**$2,040.47**	**$434.94**	**$2,475.41**
Closure Costs	**$5.85**	**$20.00**	**$25.85**
Royalties	**$115.00**	**$20.08**	**$135.08**
Pre-Tax NPV8% (includes ADEX)	**$540.56**	**$128.21**	**$668.77**
Pre-Tax IRR (includes ADEX)	**33.20%**	**106.59%**	**39.57%**
Taxes	**$328.26**	**$52.47**	**$381.04**
Post-Tax NPV8% (includes ADEX)	**$421.39**	**$93.77**	**$516.15**
Post-Tax IRR (includes ADEX)	**32.80%**	**88.71%**	**38.37%**

24.2 INCREASING CUT OFF VALUE OPPORTUNITY

There may be an opportunity to optimize the cut-off value and focus mining activities on less tonnes at higher grade for the underground mine. Third party processing and transportation costs combine to represent $40.38 per short ton mined, so removing low margin stopes has the potential to increase value. In addition, preliminary studies show higher recoveries may be achieved from higher grade feed material, creating a revenue multiplier effect by increasing the cut off value. This strategy has the potential to increase the overall margin per tonne mined and is particularly relevant early in the mine life when capital expenditures peak. Figure 24-1 illustrates the continuity of the resource at NSR cut-off values of $123.50, $140.00 $160.00, and $180.00 per tonne. Table 24-3 summarizes the PEA Potentially Mineable Resource, and the total mineral resource at these cut offs. The PEA Potentially Mineable Resource is derived from the total resource available (Scenario 1 Total Resource) at the same cut off value, with the application of stope design, including planned dilution, removal of pillars against historical workings and modifying factors including unplanned dilution and recovery. Scenarios 2-4 summarize the total in-situ resources at the various cut off values that are available for evaluation, stope design and conversion to Potentially Mineable Resource. Preliminary evaluation suggests that increasing the cut of value could reduce the tonnes mined while retaining a significant portion of the NSR value.

The mining sequence could also be optimized to focus on the higher-grade material early in the mine life which may also improve project NPV. Lower grade areas could potentially be mined later in the mine life with minimal capital expenditure and benefit from lower processing costs with a more flexible flowsheet at a future Shakespeare processing facility, once constructed. Figure 24-2 illustrates an area on the 400 ft level of the Intermediate Zone and 101 FW Zone where high-grade stopes could be sequenced earlier in the mine life. In some cases, large stopes could be redesigned into two stopes, and the higher-grade portion of the stope sequenced prior to mining the lower grade portion.

There is an opportunity to further evaluate cut-off and sequence optimization and integrate the process with a better understanding of the resource once the models are updated with new drilling results prior to the project advancing to Pre-Feasibility Study (PFS).

Table 24-3: Increased COG Evaluation Scenarios

	PEA Potentially Minable Resource	Scenerio 1 Total Resource >$123 NSR	Scenerio 2 Total Resource >$140 NSR	Scenerio 3 Total Resource >$160 NSR	Scenerio 4 Total Resource >$180 NSR
Cut-off NSR	**$ 123**	**$ 123**	**$ 140**	**$ 160**	**$ 180**
Tonnes	20,102,605	23,644,059	19,006,469	14,963,426	11,993,772
NSR/Tonne	$ 179	$ 233	$ 258	$ 288	$ 317
Total NSR	$ 3,599,700,648	$ 5,516,852,314	$ 4,908,936,802	$ 4,304,826,510	$ 3,801,461,578
Ni%	0.62	0.90	0.99	1.10	1.21
Cu%	0.55	0.78	0.86	0.94	1.02
Co%	0.02	0.03	0.03	0.04	0.04
Pt g/t	0.57	0.71	0.78	0.85	0.93
Pd g/t	0.62	0.78	0.86	0.95	1.05
Au g/t	0.34	0.42	0.45	0.50	0.54
TPM g/t	1.53	1.90	2.08	2.30	2.52
Tonnes variance		100%	80%	63%	51%
NSR Variance		100%	89%	78%	69%

Figure 24-1: Resource Continuity at Various NSR Cut-Off Values

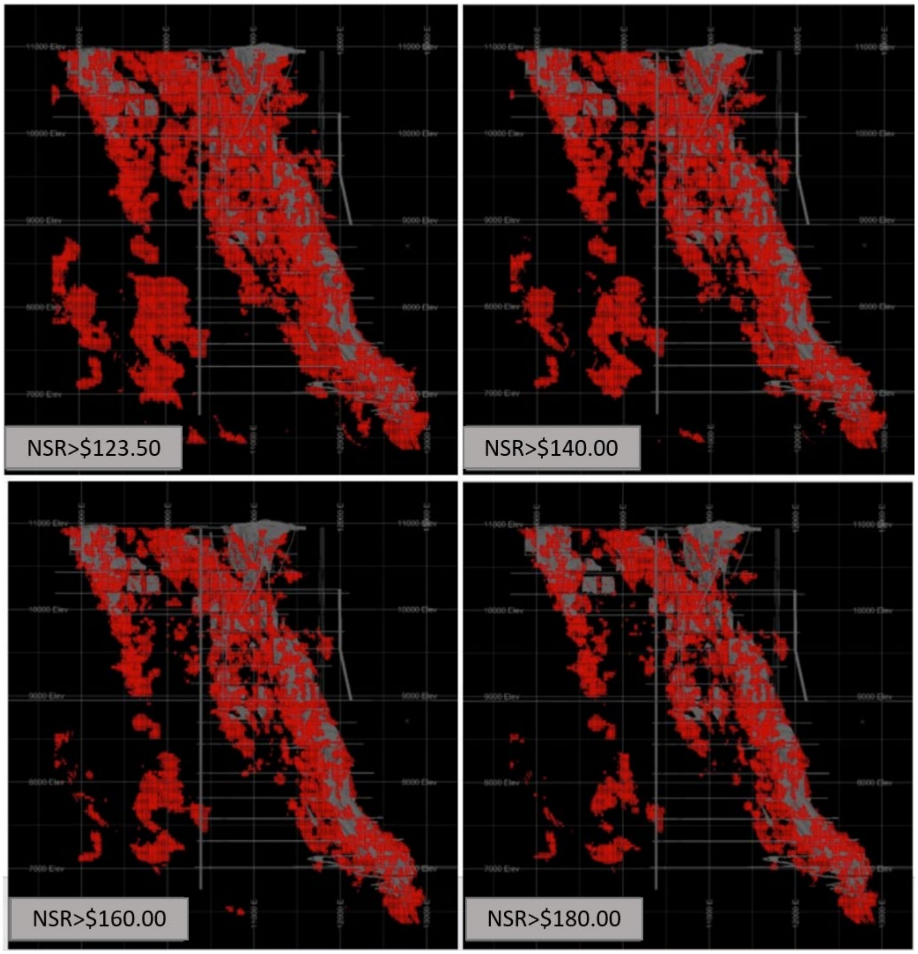
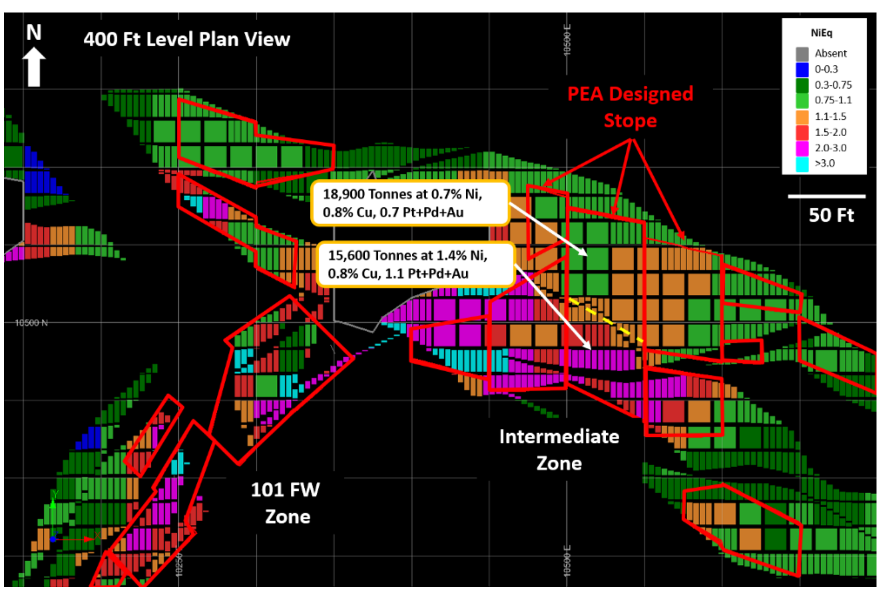


Figure 24-2: PEA Stope Design and Sequencing Opportunity



25 INTERPRETATION AND CONCLUSIONS

25.1 GEOLOGY AND EXPLORATION

Ni-Cu-Platinum Group Elements (PGE) deposits in Sudbury occur within the Sudbury Structure that formed as a result of a major early Proterozoic meteorite impact 1,850 million years ago (Ames and Farrow, 2007). The property straddles the South Range of the SIC and hosts part of a large trough structure at the base of SIC, which contains a number of previously mined ore deposits including Crean Hill Main Zone, Crean Hill Intermediate Zone, Crean Hill West Zone, Ellen Mine, and Lockerby Mine, each sitting in embayments (terraces) within the larger trough. Deposits of the Crean Hill Project include Contact Type and Footwall Type deposits, with the majority of historical mining focused on the contact Ni-Cu ore zones. From 1906-2002 a total of 20,370,000 tonnes of ore grading 1.09% Cu, 1.31% Ni, 1.56 g/t TPM was produced from the Main, Intermediate and 9400 zones.

Magna has completed a total of 12,180 m over 68 surface diamond drill holes between Q4 2022 (November) and Q3 2023. Drilling to date has been designed to enhance Magna's understanding of the near surface Intermediate, 101 FW, and 109 FW zones, both along strike and down-dip of historical mining areas. Magna diamond drilling to date has continued to confirm the grade and continuity of the current Mineral Resource Estimates as well as expand on the known mineralized zone.

25.2 MINERAL RESOURCE ESTIMATE

Completion of the updated MREs for the property involved the assessment of a drill hole database, which included all data for surface drilling completed through the end of 2017, as well as 3D mineral resource models (resource domains), 3D models of all mined-out areas (open pit and underground), 3D models of cross-cutting dykes, a recent topographic surface, and available written reports.

The Inverse Distance Squared (ID2) calculation method—restricted to mineralized domains—was used to interpolate grades for Ni (%), Cu (%), Co (%), Pt (g/t), Pd (g/t), and Au (g/t) into block models.

Indicated and inferred mineral resources are reported in Table 25-1. The current MRE considers that the Project's deposits may be mined by Open Pit and underground mining methods.

Table 25-1: Denison Deposit In-Pit (A) and Underground (below-pit) (B) Mineral Resource Estimate, August 19, 2022

Cut-off Grade	Tonnes	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	NiEq %
In-Pit Mineral Resource Estimate								
Indicated								
0.3% NiEq	16,760,000	0.53	0.49	0.02	0.48	0.37	0.25	1.08
Inferred								
0.3% NiEq	434,000	0.43	0.49	0.02	0.29	0.14	0.07	0.82
Underground (Below-Pit) Mineral Resource Estimate								
Indicated								
1.1% NiEq	14,531,000	0.96	0.84	0.03	0.88	1.02	0.54	2.07
Inferred								
1.1% NiEq	1,170,000	0.61	0.46	0.02	0.64	1.09	0.21	1.41

1. The classification of the current MRE into Indicated and Inferred is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.
2. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
3. All resources are presented undiluted and in situ, constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction.
4. Mineral resources which are not mineral reserves do not have demonstrated economic viability. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that most inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.
5. It is envisioned that parts of the Denison deposit may be mined using Open Pit mining methods. In-pit mineral resources are reported at a cut-off grade of 0.3% NiEq within a conceptual pit shell.
6. The results from the pit optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the property. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.
7. Underground (below-pit) mineral resources are estimated from the bottom of the pit and are reported at a base case cut-off grade of 1.1% NiEq. The underground mineral resource grade blocks were quantified above the base case cut-off grade, below the constraining pit shell and within the constraining mineralized wireframes. At this base case cut-off grade the deposit shows good deposit continuity with limited orphaned blocks. Any orphaned blocks are connected within the models by lower grade blocks.
8. Based on the size, shape, location, and orientation of the Denison deposit, it is envisioned that the deposit may be mined using longhole open stoping (a bulk mining method that has long been utilized in the Sudbury region).
9. High grade capping was done on 10 ft (3.05 m) composite data.
10. Bulk density values were determined based on physical test work from each deposit model and waste model.
11. NiEq Cut-off grades are based on metal prices of $8.50/lb Ni, $3.752/lb Cu, $22.00/lb Co, $1000/oz Pt, $2000/oz Pd and $1,750/oz Au and metal recoveries of 78% for Ni, 95.5% for copper, 56% for Co, 69.2% for Pt, 68% for Pd and 67.7% for Au.
12. The in-pit base case cut-off grade of 0.3% NiEq considers a mining cost of US$2.50/t rock and processing, treatment and refining, transportation and G&A cost of US$38.00/t mineralized material, and an overall pit slope of 55 degrees. The below-pit base case cut-off grade of 1.1% NiEq considers a mining cost of US$80.00/t rock and processing, treatment and refining, transportation, and G&A cost of US$42.50/t mineralized material.
13. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

To complete the MRE for the property, a database comprising a series of comma delimited spreadsheets containing surface and underground drill hole information was provided by Magna. The database included hole location information (local grid coordinates, in feet), survey data (final depth in feet), assay data (from and to in feet), lithology data, and specific gravity data. The data in the assay table included assays for Ni (%), Cu (%), Co (%), Pt (g/t), Pd (g/t), and Au (g/t), as well as Ag (g/t), Rh (ppm), S (%) and Fe (%). Note that not all assay samples had values for Pt, Pd, Au, Ag, or Rh. Ag and Rh were the least-analysed elements and are not included in the MRE.

After reviewing the database, data were imported into GEOVIA GEMS version 6.8.3 (GEMS) for statistical analysis, block modeling, and resource estimation.

The database used for the current MRE comprises data for 3,836 surface and underground drill holes completed within the deposit area, which total 1.57 million ft (478,000 m). The database totals 89,257 assay intervals for 622,082 ft (189,611 m).

The database was checked for typographical errors in drill hole locations, down-hole surveys, lithology, assay values, and supporting information on source of assay values. Overlaps and gapping in survey, lithology, and assay values in intervals were checked. Gaps in the assay sampling and unsampled elements were assigned a grade value of 0.0001 for Co, Pt, Pd, and Au.

The MREs for the property are prepared and disclosed in compliance with all current disclosure requirements for mineral resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). The classification of the current MREs into Inferred is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves, including the critical requirement that all mineral resources “have reasonable prospects for eventual economic extraction”.

The general requirement that all mineral resources have “reasonable prospects for eventual economic extraction” implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade, considering extraction scenarios and processing recoveries. To meet this requirement, the author considers the Denison deposit mineralization amenable for open pit and underground extraction.

To determine the quantities of material offering reasonable prospects for economic extraction by an open pit, Whittle™ and reasonable mining assumptions were used to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be reasonably expected to be mined from an open pit. The pit was optimized by SGS. The pit optimization parameters used are summarized in Table 25-1. A Whittle™ pit shell at a revenue factor of 1.0 was selected as the ultimate pit shell for the purposes of this MRE. The corresponding strip ratio is 10.6:1 and reaches a maximum depth below surface of approximately 1,320 ft (402 m) in the east and 1,250 ft (381 m) in the west. The optimized pit is limited to the property boundary.

The from the pit optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the property. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade. A selected base-case cut-off grade of 0.3% NiEq was used to determine the in-pit MRE for the Denison deposit.

To determine the quantities of material offering “reasonable prospects for economic extraction” by underground mining methods, reasonable mining assumptions were made to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be “reasonably expected” to be mined from underground. Based on the size, shape, and orientation of the deposit, it is envisioned that the deposit may be mined using the longhole open stoping mining method, a bulk mining method that has long been utilized in the Sudbury region. A selected base-case cut-off grade of 1.1% NiEq was used to determine the below-pit MRE for the Denison deposit. The below-pit MRE is limited to a depth of approximately 4,500 ft (1,371.6 m) below surface.

The underground resources are presented undiluted and in situ (no minimum thickness), constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction. There are no underground mineral reserves reported at this time.

There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading. The author is not aware of any known mining, processing, metallurgical, environmental, infrastructure, economic, permitting, legal, title, taxation, socio-political, or marketing issues—nor any other relevant factors not reported in this technical report—that could materially affect the current MRE.

25.3 MINING

The inferred and indicated mineral resources in the Crean Hill Project resource model were used as the basis for a PEA. The PEA considers a mine plan that includes both an underground mine and open pit mine producing a combined 20.1 million tonnes at an average NSR of $179.07 per tonne over a 15-year mine life. The average grade of mined material is 0.62%Ni, 0.55%Cu, 0.02%Co, 0.57gpt Pt, 0.62gpt Pd, and 0.34gpt Au.

The proposed open pit produces 3.8 million tonnes of mineralized material at an average NSR of $131.05 per tonne. Approximately 29.3 million tonnes of waste rock are removed from the pit resulting in a strip ratio of 7.7. The proposed open pit is mined by a contractor.

The underground mine uses typical development and production methods and equipment. The proposed mining methods are longitudinal longhole stoping and mechanized cut and fill, with cemented and unconsolidated rockfill used for backfill. The existing flooded mine workings require dewatering. A new ramp from the surface will be developed to support an initial ADEX program followed by Phase 1 mining to 2000L using trucks to haul to surface. During Phase 1, the existing shaft will be reconditioned and re-equipped for production and servicing the mine during Phase 2 production to 4350L. The production rate during Phase 1 will be approximately 2,200 tonnes (2,400 short tons) per day and will increase to 3,600 tonnes (4,000 short tons) per day during Phase 2.

The underground mine produces 16.3 million tonnes of mineralized material at an average NSR of $190.36 per tonne over a 15-year mine life.

There is some risk around the accuracy and completeness of the historic voids in the mine model. If there are historic stope voids that have been mined but are not shown as being mined in the resource model and subsequent mine design, the mine plan could be over-stating production. This risk is partially mitigated in the underground mine design by leaving a 5 m (18.4-foot) thick perimeter pillar around all historic stope voids that are in the model. None of this perimeter pillar is mined in Blocks 1, 2, and 3. In Block 4, where there is greater confidence in the void model and some stopes are known to be filled with consolidated backfill, only 20% of cut-and-fill shapes generated in the pillar are included in the mine plan.

25.4 METALLURGY AND PROCESSING

The test work performed to date on 109 FW demonstrates similar grinding and flotation performance to other Sudbury basin resources. Before entering into an ore sale agreement with a local mining company, mineralogical and metallurgical testing will be performed on samples that represent the materials that will be processed, according to the local mining company's selection and testing protocols, to confirm metal recovery to concentrate.

25.5 INFRASTRUCTURE

The Crean Hill Project is well positioned with existing road access, nearby power infrastructure, nearby water sources, an active water treatment facility, and a site well-prepared for operations by past mining activity. The proximity to Sudbury and surrounding communities provides access to a skilled workforce that can commute to the site during all phases of activity. Similarly, access to the numerous service providers, vendors, and supply chains eliminates the need for storing a large quantity of consumables on site.

25.6 PERMITTING

The Project is regulated through all phases by both federal and provincial agencies. Vale has been conducting environmental studies at the property in accordance with regulatory requirements for decades and data collection is ongoing. Cumulatively, environmental studies generally cover the following areas of study.

- Surface water quality and hydrology
- Ground water quality and hydrogeology
- Geochemistry
- Terrestrial and aquatic biology

Magna will complete a gap analysis to determine if supplemental studies are warranted to support the permitting process for the development of the Project. Environmental studies characterize the biophysical environment prior to development of the Project. Technical studies to date have not identified biophysical or cultural heritage values that pose a material constraint to development. The remaining environmental liabilities at the property are primarily the chemical stability risks associated with mine rock piles and decant water from the open pit and underground mine work.

The average production capacity for the mine is anticipated to be below an assumed 5,000 tonne per day threshold in Section 18 of the Physical Activities Regulations made under the Impact Assessment Act.

The Project is not required to complete a federal impact assessment. However, under Section 9(1) of the Impact Assessment Act, the Minister of Environment and Climate Change may designate a physical activity that is not prescribed by the Physical Activities Regulations if, in their opinion, either the carrying out of that physical activity may cause adverse effects within federal jurisdiction or adverse direct or incidental effects, or public concerns related to those effects warrant the designation.

The Project will need to complete the class EA processes listed below, which may be amended in the future by the provincial government.

- Class EA for Resource Stewardship and Facility Development, in accordance with MNR (2003), in advance of issuing permits for Crown timber harvesting, water crossings (>5km^2 watershed area), occupying and constructing on Crown owned surface rights, online dams and aggregate pit development.
- Class EA for any land tenure decisions, administered by Ministry of Mines
- Category B Class EA is required under the *Electricity Projects Regulation* (O. Regulation 116/01) for the use of diesel generators ($\geq$1MW and <5MW cumulative capacity).
- An evaluation will be required for potential EA requirements under the *Electricity Projects Regulation* (O. Regulation 116/01) and possible approvals from the Ontario Energy Board for potential upgrades to electricity and natural gas service.

The permits that Magna anticipates will be required by the federal government are listed in Table 20-1, by the provincial government in Table 20-2, and the municipal government in Table 20-3. All are subject to change, depending on final project details.

Two permits that are currently in place for the property are the following.

- Crean Hill Mine Closure Plan
- Sewage ECA 6763-9ZXQWA for the Crean Hill and Ellen Pit Wastewater Treatment Facility

It is possible that these permits can be amended to support the Project, and Magna will consult with government agencies as planning progresses to confirm permit requirements.

25.7 CAPITAL AND OPERATING COSTS

The capital and operating costs consider the direct and indirect costs associated with the ADEX program, mine development and construction, and mine production. The costs for underground development and production are based on mining quantities from the mine model with appropriate allowances applied and unit rates built from recent experience with other projects and recent labour and database materials pricing.

Capital purchases and installation construction costs are based on quotes and estimates sourced from other recent projects with similar installations. The capital costs include 35% contingency on capital during the ADEX program and Project Capital period to reflect the level of engineering and reliance on assumptions that are typical of a PEA.

25.8 ECONOMIC ANALYSIS

The Crean Hill Project combined Underground and Open Pit mining operation demonstrates a potential post-tax NPV (discounted 8%) of $230.4 million, IRR of 23.4%, and 3.3-year payback period of initial project capital during a 15-year mine life (from the start of ADEX).

26 RECOMMENDATIONS

The QPs authoring this PEA caution that the PEA is preliminary in nature and includes inferred resources that are too speculative geologically to have economic considerations applied to them. There is no certainty that this PEA will be realized. This PEA should only be treated as an initial indication of the potential viability of the Crean Hill Project mineral resources.

Based on the results of the PEA, the QPs recommend the Crean Hill Project advances to a Pre-Feasibility Study (PFS). The PFS should be based on indicated and measured resources in an updated resource model that includes additional diamond drilling information. The purpose of the PFS is to advance the engineering of the mine designs and infrastructure designs, investigate unknowns and assumptions, update the cost estimate to AACE Class 4, and support a mineral reserve estimate for the Project. The PFS should also further investigate the potential synergies with other Magna projects and the results of the PFS should be disclosed in an NI 43-101 Technical Report.

26.1 GEOLOGY

The Crean Hill Deposit contains within-pit and underground Indicated and Inferred Mineral Resources that are associated with well-defined mineralized trends and models. The deposit is open along strike and at depth.

Given the prospective nature of the deposit, it is the author's opinion that the Project merits further exploration and that a proposed plan for further work by Magna is justified. A proposed work program by Magna will help advance the Project and will provide key inputs required to evaluate the economic viability of the Project at a PFS level.

The author is recommending Magna conduct further exploration, subject to funding and any other matters which may cause the proposed exploration program to be altered in the normal course of its business activities or alterations which may affect the program because of the exploration activities themselves.

The total cost of the recommended work program by Magna is estimated at C$3.42 million to C$4.22 million (Table 26-1).

To date, Magna has completed 68 surface diamond drill holes between Q4 2022 and Q3 2023 for a total of 12,180 m. Drilling to date has been designed to enhance Magna's understanding of the near surface Intermediate, 101 FW, and 109 FW zones both along strike and down-dip of historical mining areas.

Magna intends to continue drilling and is planning an additional 12,000 m to 15,000 m of diamond drilling during the H2 2023 and end of 2024. Drilling will continue to focus on upper levels, roughly above the 2,000 ft mine level to further define and expand on known resources, as well as deeper drilling exploring for new FW zones. Underground diamond drilling will commence as the ADEX program progresses and drilling platforms become available.

A surface bulk sample is recommended on the 109 FW zone to better understand continuity of mineralization and provide material for additional metallurgical test work.

The 2023/2024 work program includes diamond drilling, geological compilation, and mineral resource estimation, as summarized in Table 26-1.

Table 26-1: Recommended 2023/24 Work Program for the Crean Hill Project

Item	Estimated Cost (CAD)
Diamond Drilling [1] (12,000 m to 15,000 m, $250/m)	$3,000,000 to $3,750,000
Geological Compilation, Revised Mineral Solid Modeling, Revised Resource Estimation [2]	$125,000 to $150,000
Surface Advanced Exploration and Bulk Sample	$300,000 to $325,000
Total:	**$3,425,000 to $4,225,000**

[1] Includes sampling cost, assaying, logging, geotechnical, drill management, core storage, travel accommodation, logging facilities, consumables, and data reporting.
[2] Includes NI43-101 Technical Reporting.

26.2 MINING

The underground and open pit mine designs presented in this PEA are a reasonable approach to mining the Crean Hill Project resources. The Mining QPs recommend advancing the mine engineering, designs, and economic evaluation to a PFS level to increase the confidence in the designs and support a mineral reserve estimate. The recommended work during the PFS is summarized in Table 26-2.

Table 26-2: Recommended Crean Hill Project Mine Engineering and Design PFS Work

Recommended work during PFS	Estimated Cost (CAD)
Investigate available geomechanical and hydrogeology information.	$200,000 to $250,000
Complete further geomechanical investigations (including drilling) and data collection and input to the design criteria.	$550,000 to $750,000
Investigate the condition of the existing shaft and work required to re-establish as a production shaft.	$100,000 to $200,000
Complete desktop trade-off studies that could include: • Investigate the accuracy and completeness of the existing mine workings in the mine model. • Mining methods. • Backfill type. • Battery Electric Vehicles. • Update NSR calculations and optimized mining cut-off NSR. • Evaluate a higher underground cut-off NSR scenario that targets higher grade material. • Timing of bringing the production shaft into service. • Timing of Open Pit versus Underground mining. • Open Pit Contractor Operated versus Owner Operated.	$160,000 to $320,000
Advance the underground mine design and development/production schedule to support an AACE Class 4 cost estimate.	$150,000 to $200,000

Recommended work during PFS	Estimated Cost (CAD)
Advance the open pit mine design and production schedule to support an AACE Class 4 cost estimate.	$100,000 to $150,000
Advance the underground and surface infrastructure designs to PFS level engineering and design to support an AACE Class 4 cost estimate.	$150,000 to $400,000
AACE Class 4 cost estimate, schedule, and financial analysis	$100,000 to $150,000
NI 43-101 Technical Report	$50,000 to $100,000
Total for Mine Engineering and Design Recommendations	**$1,560,000 to $2,520,000**

26.3 MINERAL PROCESSING AND RECOVERY

Based on the results of the test work completed at the time of publication of this report, the following are recommended by the mineral processing QP to complete further testing:

1. To complete the evaluation of selling ore to an existing mining company.
    a. Evaluation of representative samples from untested zones. Estimated cost $50,000.
    b. Evaluation of blending to enhance precious metal recovery. Estimated cost $50,000.
2. To evaluate other opportunities to enhance value.
    a. Evaluation of impact of gravity separation on coarse primary grind. Estimated cost $50,000.
    b. Evaluation of ore sorting through testing of representative samples. Estimated cost $300,000.
    Evaluation of processing through the potential Shakespeare Mill with necessary plant modifications to handle additional concentrate. Estimated cost $100,000

26.4 ENVIRONMENTAL AND PERMITTING

The Environmental and Permitting QP recommends completing the following environmental and permitting work during a PFS.

- Continue to stay apprised of Vale Crean Hill/Ellen pits closure activities.
- Confirm future water treatment capacity at the Crean Hill treatment plant to accept open pit and underground working, dewatering, and mine redevelopment runoff.
- Maintain minimum environmental setbacks/buffers from waterbodies/watercourses.
- Confirm the fish habitat status of on-site waterbodies and avoid deposition of mine waste in waters frequented by fish.
- Conduct wildlife and terrestrial vegetation surveys throughout the Property.
- Continue to include the entire Crean Hill/Ellen Pits closure boundary in the project boundary and restrict mine area expansion to less than 50% of existing/previous mine area (inclusive of Ellen Pits).
- Confirm the production history that will establish the threshold for triggering a federal impact assessment and consider this threshold in the mine production rate.
- Continue geochemical characterization of mineralized material, waste rock, and overburden.

- Continue engagement with indigenous stakeholders and the local communities.
- Commence regulatory engagement towards mine re-opening.

The estimated cost of the environmental and permitting recommendations is $85,000 for a PFS.

27 REFERENCES

Ames, D.E., and Farrow, C.E.G. 2007. *Metallogeny of the Sudbury mining camp, Ontario, in Goodfellow, W.D., ed., Mineral Deposits of Canada: A Synthesis of Major Deposit-Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods*. Mineral Deposits Division, Geological Association of Canada, *Special Publication No. 5*, 329—350.

Armitage, A. 2022. *Technical Report on the Mineral Resource Estimate for the Denison Deposit, Denison Project*, Sudbury, Ontario Canada. SGS.

Armitage, A. 2022. *Mineral Resource Estimate Update for the Denison Ni-Cu-PGE Sulphide Deposit, Denison Project*. Sudbury, Ontario, Canada: Magna Mining Inc., 137.

Baker, A. and Hoffman, D. 2017. *Denison 9400 Mineral Resource Estimation*. Internal Report, Lonmin Canada Inc.

Baker, A., and Hoffman, D. 2015. *Denison 109 Footwall Mineral Resource Estimation*. Internal Report, Lonmin Canada Inc.

Bleeker, W., Kamo, S., Ames, D., and Smith, D. 2014. *New U-Pb ages for some key events in the Sudbury area, including the Creighton Granite and Joe Lake metagabbro. GAC-MAC Joint Annual Meeting, Abstracts Vol. 37*, 33.

Capri, L. 2016. Precious Metal Deportment in Six Composite Samples from the Denison 9400 FW Zone.

Card, K.D., Innes, D.G. and Debicki, R.L. 1977. *Stratigraphy, sedimentology and petrology of the Huronian Supergroup in the Sudbury–Espanola area*. Ontario Division of Mines, *Geoscience Study 16*, 99.

Dressler, B.O. 1984. *General geology of the Sudbury area; in The Geology and Ore Deposits of the Sudbury Structure*. Ontario Geological Survey, *Special Publication No. 5*, 205-222.

Eckstrand, O.R., and Hulbert, L.J. 2007. *Magmatic nickel-copper-platinum group element deposits, in Goodfellow, W.D., ed., Mineral Deposits of Canada: A Synthesis of Major Deposit Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods*. Mineral Deposits Division, Geological Association of Canada, *Special Publication No. 5*, 205-22.

Environment and Climate Change Canada (ECCC). 2023. "Canadian Climate Normals 1981-2010 Station Data - Climate." ECCC.

Farrow, C.E.G., Everest, J.O., King, D.M., and Jolette, C. 2005. *Sudbury Cu (-Ni)-PGE systems: refining the classification using McCreedy West Mine and Podolsky Project case studies*. Mineralogical Association of Canada, *Short Course 35*, 163-180.

Ford, F. et al. 2010. *Process Mineralogy of PGMs from Denison 109FW Zone Sample 1244*. Internal Report, Vale.

Gould, C. et al. 2022. *Technical Report Summary, Sudbury Property, Ontario Operations, Canada*. Vale.

Kelly, A. 2018. *Denison Prefeasibility Study*. Blue Coast Research PJ5219.

Kelly, A. 2020. *Gravity Recovery and Flotation Optimization*. Blue Coast Research PJ5313.

Kjarsgaard, I., and Ames, D.E. 2010. *Ore Mineralogy of Cu-Ni-PGE Deposits in the North Range Footwall Environment, Sudbury, Canada: 11th International Platinum Symposium abstracts*. Miscellaneous Release-Data 269, Ontario Geological Survey.

Krstic, S. 2015 and 2017. Vale Base Metals Resource Management Group (Baker and Hoffman). *A Summary of Vale Core Logging, Sampling, Sample preparation, analysis, QA/QC Practices and Assay Validation Procedures.*

Lee, A. et al. 2012. *Denison 109 Zone MinMet 2011*. Internal Report, Vale.

Lee, A. et al. 2010. *Phase 2 Metallurgical Evaluation of Denison PGM*. Internal Report, Vale.

Lightfoot, P.C. 2017. *Controlling factors on the distribution of low-S, high-3E mineralization in the Denison 9400 Zone and surrounding footprint*. Geosciences Report, Lonmin Canada Inc.

Lightfoot, P.C. 2016. *Nickel Sulphide Ores and Impact Melts: Origin of the Sudbury Igneous Complex*. Elsevier. In the Press.

Lightfoot, P.C., and Farrow, C.E.G. 2002. *Geology, Geochemistry, and Mineralogy of the Worthington Offset Dike: A Genetic Model for Offset Dike Mineralization in the Sudbury Igneous Complex*. *Economic Geology 97*: 1419-1446.

Lunder, P.J., and Pakalnis, R. 1997. *Determining the Strength of Hard Rock Mine Pillars*. Bull. Can. Inst. Min. Metall. 90: 51-55.

Manqui, X. et al. 2010. *Mineralogy and Metallurgy of Denison PGM Ore*. Internal Report, Vale.

MEND. 2009. *Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials*. MEND Report 1.20.1.

Ministry of Environment (MOE). 1994. *Deriving Receiving Water Based, Point Source Effluent Requirements for Ontario Waters*.

Ministry of Environment (MOE). 2005. *PTTW Manual*.

Ministry of Natural Resources (MNR). 2003. *A Class Environmental Assessment for MNR Resource Stewardship and Facility Development Projects*.

Ministry of Natural Resources (MNR). 1995. *Environmental Guidelines for Access Roads and Water Crossings*.

Naldrett, A.J. and Hewins, R.H. 1984. *The Main Mass of the Sudbury Igneous Complex; in The Geology and Ore Deposits of the Sudbury Structure*. Special Volume 1, Ontario Geological Survey, 235-251.

Péntek, A., Molnár, F., Watkinson, D.H., and Jones, P.C. 2008. *Footwall-type Cu–Ni–PGE mineralization in the Broken Hammer area, Wisner Township, North Range, Sudbury Structure*. Economic Geology 103: 1005-1028.

Senes. 2004. Table B4. “Acid Base Accounting (ABA) Data” in *Draft Report on Geochemical Modelling of Crean Hill Mine Backfilled Pits*. Senes Consultants.

SGS Geological Services. 2022. *Mineral Resource Estimate for the Denison NI-CU-PGE Sulphide Deposit, Denison Project*. Subdury, Ontario, Canada.

Soil Survey of Canada. 1983. *Soils of Sudbury Area, Ontario*. Soils Survey Map No. 49.

SRK Consulting (Canada). 2020. *Preliminary Economic Assessment for the Denison Base Metal Project*. Project number: 5CL007.000, Lonmin Canada Inc., Lively, Ontario, Canada.

SRK Consulting (Canada). 2006. *Property Summary Memo*. INCO Technical Services.

Stewart, M.C. and Lightfoot, P.C. 2010. *Diversity in Platinum Group Elements (PGE) Mineralization at Sudbury: New Discoveries and Process Controls*. 11th International Platinum Symposium.

Tetra Tech Wardrop. 2012. *Denison Project Prefeasibility Open Pit Geotechnical Evaluation, Document No. 1102071700-REP-R0001-01*.

Tuba, Gy., Molnár, F., Watkinson, D.H., Jones, P.C., and Mogessie, A. 2010. *Hydrothermal vein and alteration assemblages associated with low sulphide footwall Cu-Ni-PGE mineralization and regional hydrothermal processes, North and East Ranges, Sudbury structure, Canada*. Society of Economic Geologists, Special Publication 15 2: 573-598.

Vale. 2022. "Crean Hill Mine Closure Plan, Part I of II." Vale.

WSP. 2020. *Denison Project Resource Review, Denison Twp., Sudbury District*. Project number 191-14115-00_RPT-01_R3, Lonmin Canada Inc.