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Endako Mine Restart Preliminary Economic Assessment

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The technical report entitled "National Instrument 43-101 Technical Report (the "**Technical Report**") for the Endako Mine Restart Preliminary Economic Assessment" (the "**PEA**") with an effective date of January 5, 2026 for Moon River Moly Ltd. The report is prepared in accordance with National Instrument (NI) 43-101 – *Standards of Disclosure for Mineral Projects* and Form 43-101F1 – Technical Report. The issue date of this report is January 5, 2026.

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1.0 EXECUTIVE SUMMARY

The Endako Mine Restart (the "**Project**" or "**Endako**"), British Columbia, Canada, Preliminary Economic Assessment (PEA) was prepared for Moon River Moly Ltd. (Moon River) of Toronto, Canada, by A-Z Mining Professionals Ltd. (AMPL), Canada. This PEA provides the proposed plan and expenditures required to restart the past producing Endako Mine (which has been on care and maintenance status since July 2015) and provides guidance to Moon River on how best to progress the Endako Mine Joint Venture to restart operations using the existing mineral resources.

This PEA assesses the potential economic viability of the Project. The cost estimates fall within the guidance on accuracy for PEAs (±40%). This report is prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and in accordance with the requirements of Form 43-101 F1. The disclosure is based on reliable information, the professional opinions of independent Qualified Persons and uses industry best practices and standardised terms.

The Endako Mine is a past producing mine with flooded open pits, an original mothballed processing facility (the "**Old Plant**") which had a processing capacity of approximately 30,000 tonnes per day, a modern 52,000 tonnes per day processing facility that has been on care and maintenance since 2015 (the "**New Plant**") and all support infrastructure and facilities in place (also on care and maintenance). Both processing facilities employed flotation recovery producing molybdenum concentrate.

The Project contemplates restart of open pit mining with potentially economic mineralisation processed in the existing on-site processing facilities. The open pit will produce approximately 26 to 27 million tonnes per year of potentially economic mineralisation. ROM potentially economic mineralisation will be crushed, ground in the SAG mill located in the New Plant, ground in ball mills located in both the New and Old Plants and will produce a molybdenum concentrate by flotation. The resulting life-of-mine (**LOM**) is 10.2 years.

The PEA results, applying the long-term three-year trailing average price of US$49.73 per kilogram (US$22.50 per pound) of molybdenum, at an exchange rate of CA$1.00 = US$0.74, indicate that a restart of the Endako Mine would result in very viable project economics.

All currency references are in 2025 Canadian Dollars, unless otherwise specified.

1.1 PROPERTY DESCRIPTION

The Endako Mine is located in the Bulkley-Nechako region of central British Columbia, approximately 190 kilometres (km) west of Prince George and about 400 km east of Prince Rupert, British Columbia. The nearest town is Fraser Lake, about 15 km east of the Mine. The Mine is located about 10 km south of Highway 16 (Yellowhead Highway) on a paved road. The village of Endako is on Highway 16 at the junction with the Endako Mine Road. The highways to Fraser Lake and Prince George are paved, high quality roads that are part of the Trans-Canada Highway system.

The Project consists of 34 legacy and converted legacy claims and 26 mineral leases covering an area of approximately 12,797.2 hectares (ha). As of August 1, 2025, information extracted from the British Columbia mineral title online database indicated all claims and leases are in good standing.

The Endako Mine is jointly owned by Moon River (25%) and Centerra Gold Inc. (Centerra) (75%) and operates through a joint venture company.

1.2 GEOLOGY

The Endako molybdenite deposit is hosted within the Endako Quartz Monzonite, which is intruded by younger Casey Alaskite toward the north and François Granite toward the south. In the Endako Mine area, Endako Quartz Monzonite has been intruded by aplite, andesite, quartz-feldspar porphyry and porphyritic granite dykes and post-ore basaltic dykes.

The Francois Lake Intrusions occur as numerous granitic intrusions of middle to late Jurassic age. The main body of the Francois Lake Intrusions, which is the host of the Endako molybdenite deposit, is a large northwesterly trending composite batholith that has been emplaced along the boundary zone between the Cache Creek and Stikine terrane of the Intermontage Tectonic Belt.

The Endako phase of the Francois Lake Intrusion forms a northwesterly, elongated body consisting of coarse-grained, dark pink to orange, biotite-hornblende granodiorite to monzogranite, sub-porphyritic with distinctive orange K feldspar phenocrysts.

The Casey phase borders the Endako phase on its northern side. It consists of fine to medium grained, dark pink, granophyric biotite monzogranite.

Structural studies that were initiated in 1973 resulted in the creation of the "Endako Vein System" concept. This model is based on the fact that a complex array of structural elements present in the Endako orebody are actually distributed along certain natural axes. When these axes are properly identified, it allows the grouping of the structural elements into natural workable units or systems. Each system, therefore, possesses a definite structural style, and because hydrothermal mineralogy is a function of structure, each system also possesses a characteristic mineralogical style.

Endako Mine personnel further split the Endako phase into 13 smaller sub-units, along with an additional fourteenth unit designated for the Casey. A Mineral Resource was calculated individually for each unit before the results were combined.

1.3 MINERALISATION

At Endako, the mineralisation consists of molybdenite with a gangue of pyrite, magnetite, minor chalcopyrite and rare bornite, bismuthinite, scheelite and specularite. The orebody consists of a series of sub-parallel or en-echelon quartz-molybdenite-pyrite veins and stockwork veins, veinlets and mineralised fractures. The increase in frequency of these veins along a preferred axis form part of the vein system concept.

Mineralisation occurs in milky white to banded or ribboned quartz veins that are often brecciated and healed by quartz and late-stage calcite and minor chalcedony. Molybdenite varies in grain size from very coarse and greasy to microscopic blue-black grains in quartz referred to as "black quartz ore". A pyrite zone lies to the south of, and adjacent to, the orebody with a transitional boundary in the immediate hanging wall of the South Basalt Fault.

Extensive hydrothermal alteration occurs within the Endako ore zone. K-feldspar bearing envelopes develop around quartz-molybdenite veins and barren quartz veins in the footwall of the deposit. Sericite envelopes, consisting of quartz, sericite and pyrite, are developed around quartz-molybdenite and quartz-magnetite veinlets in the orebody, and quartz-pyrite veins in the pyrite zone. Argillic alteration (kaolinization) is pervasive throughout the orebody, ranging from weak to intense.

1.4 MINERAL RESOURCES AND RESERVES

The resource data and block model, completed by Mr. F. Bakker, P.Geo. of AMPL, utilized electronic data provided by Centerra and backup physical documentation stored at the Endako Mine Site. This resource model was used for determining the mineral resources estimate and to undertake open pit optimisation, potentially mineable resource estimation and completion of a mine production plan.

Table 1.1, below, presents the mineral resource estimate for the Endako Mine at various cut-off grades.

The highlighted resource at a cut-off grade of 0.040% MoS_2, is the long-term potentially economic mineralisation that could be available for mining. This cut-off grade was used in determining the mining plan, which reflects the mining and processing rate of approximately 27 million tonnes per year at an overall LOM operating cost of $11.84 per tonne. The 0.040% MoS_2 cut-off was chosen as it has an in-situ value of $12.14 per tonne, which is slightly above the LOM mining cost. (Grade/100 × 1,000 kilogram per tonne × conversion to Mo (0.5994%) × Mill Recovery (75.7%) × $49.73 (price per kilogram) × 1.35 (US$ Exchange Rate) = $12.14 per tonne).

Past mining has created a series of surface stockpiles of potentially economic mineralisation, with material added and removed from the stockpiles tracked until the mining and processing was suspended in 2014. This stockpile material will be used to commission the combined processing plants.

It should be noted that the PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. Metallurgical and cost projections are to a PEA level of accuracy. Therefore, there is no guarantee that the economic projections contained in this PEA will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability

There are no Mineral Reserves for the mine.

TABLE 1.1
MINERAL RESOURCE ESTIMATE

Zone		Cut-off % MoS_2	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
			Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Total	>=	0.010	237,413,000	0.047		435,641,000	0.049		673,054,000	0.048		164,564,000	0.038
Total	>=	0.015	206,183,000	0.052		409,652,000	0.052		615,834,000	0.052		144,091,000	0.037
Total	>=	0.020	183,642,000	0.056		382,707,000	0.054		566,349,000	0.055		128,689,000	0.040
Total	>=	0.025	157,962,000	0.062		347,564,000	0.057		505,527,000	0.059		112,503,000	0.043
Total	>=	0.030	138,289,000	0.067		311,767,000	0.061		450,056,000	0.063		93,871,000	0.046
Total	>=	0.035	117,593,000	0.073		271,696,000	0.065		389,288,000	0.067		76,928,000	0.050
Total	>=	0.040	100,673,000	0.079		234,981,000	0.069		335,654,000	0.072		60,127,000	0.054
Total	>=	0.045	85,723,000	0.086		187,826,000	0.074		273,550,000	0.078		45,770,000	0.060
Total	>=	0.050	73,121,000	0.093		158,985,000	0.079		232,107,000	0.084		34,961,000	0.066
Total	>=	0.055	62,662,000	0.100		132,436,000	0.085		195,096,000	0.090		27,753,000	0.072
Total	>=	0.060	54,246,000	0.106		112,264,000	0.090		166,509,000	0.095		22,864,000	0.077
Total	>=	0.065	46,871,000	0.113		93,091,000	0.096		139,960,000	0.102		18,903,000	0.083
Total	>=	0.070	40,936,000	0.120		78,354,000	0.101		119,289,000	0.108		15,875,000	0.087
Total	>=	0.075	35,776,000	0.127		64,680,000	0.107		100,455,000	0.114		13,276,000	0.092
Total	>=	0.080	31,269,000	0.134		54,004,000	0.113		85,270,000	0.121		11,042,000	0.097

Source: *AMPL, 2025*

1.5 PROJECT RESTART PLAN

The Endako Mine restart plan comprises:

1. Mining measured and indicated potentially economic mineralisation, located mainly in the walls and floors, of the existing Endako and Denak open pits.
2. Rebuilding existing blast hole drills and wire rope shovels from the past operation and leasing major mining equipment and purchasing smaller support equipment for the mine.
3. Relocating the existing primary crusher and building a second new primary crusher facility to both be outside of the new ultimate open pit limits.
4. Adding pumping capacity to split the mill feed between the New Plant and the Old Plant ball mills and add extra rougher flotation capacity to the New Plant to allow for the planned higher throughput.
5. Refurbishing a portion of the Old Plant grinding circuit to add an extra 20,000 tonnes per day to 25,000 tonnes per day grinding capacity and connecting this grinding circuit to the New Plant flotation circuit.
6. Refurbishing of major equipment and refurbishing or replacing smaller equipment components (as required) in the New and Old Plants.
7. Construction of a new concentrate dewatering and drying circuit and building.
8. Construction of a tailings sand plant to produce cycloned tailings to facilitate tailings dams height and storage capacity increases, to meet the restart mine plan.
9. Refurbishing and/or upgrading, as required, of the site infrastructure buildings, facilities and services.
10. Construction of a water treatment plant to treat all contact water (from disturbed areas of the mine) that will not be recycled in the mining and processing operations.
11. Reconfiguring and installation of water pipelines to store and remove water (from the disturbed areas of the mine) that will not be recycled in the mining and processing operations.
12. Construction of water pipelines from TP-1 to the water treatment plant and from the water treatment plant to the discharge to the environment location, on the Endako River.
13. Sale of molybdenum concentrate to domestic and international smelters.

1.6 MINING

Mining would employ open pit techniques using conventional rubber tired, diesel-powered mobile equipment, track mounted drills and wire rope shovels.

The LOM potentially economic mineralisation production is scheduled at approximately 26 to 27 million tonnes per year A total maximum mining rate of 45.4 million tonnes per year of potentially economic mineralisation and waste over the LOM operations is scheduled. The LOM strip ratio is favourable at 0.68 tonnes of waste per 1 tonne of potentially economic mineralisation. The peak strip ratio occurs in Year 1 at 0.97 and then remains relatively constant at approximately 0.69.

The mine schedule is based on the optimised pit shell with mining recovery and mining dilution rates of 95% and 5%, respectively, incorporated.

Mining will be performed on a 24 hour, 7 days per week basis. Mining equipment will be a combination of leased and owned.

The potentially economic mineralisation from the open pit optimisation was used to develop the mine plan, which extracted 273 million tonnes at an average grade of 0.075% MoS_2 (0.045% Mo) after dilution and mining losses.

1.7 PROCESSING

Recovery of a molybdenum concentrate from mined ore will be achieved using the refurbished existing processing plant's facilities. The addition of new processing equipment is required for concentrate leaching, dewatering and drying. The processing plant will have a capacity to treat 75,000 tonnes per day or 27 million tonnes per year.

The existing in-pit primary crusher and a new second crusher, located close to the New Plant, will process the potentially economic run of mine (**ROM**) mineralisation. The crushed potentially economic mineralisation will be ground in the SAG mill located in the New Plant followed by grinding in ball mills located in both the New Plant and the Old Plant.

The grinding products from the Old Plant and the New Plant will be combined at the beginning of the New Plant flotation circuits. Concentrate will be produced by rougher/scavenger flotation, primary concentrate regrinding, first cleaner/scavenger flotation, secondary concentrate regrinding, secondary cleaner flotation, and final concentrate thickening.

Final concentrate leaching and product dewatering and drying will be performed in a new facility constructed on the footprint of the existing ultra-pure plant (to be demolished).

A generalized flowsheet for the reconfigured Endako processing plant is presented in Figure 1.1, below.

The forecast average processing plant molybdenum recovery is 75.7%

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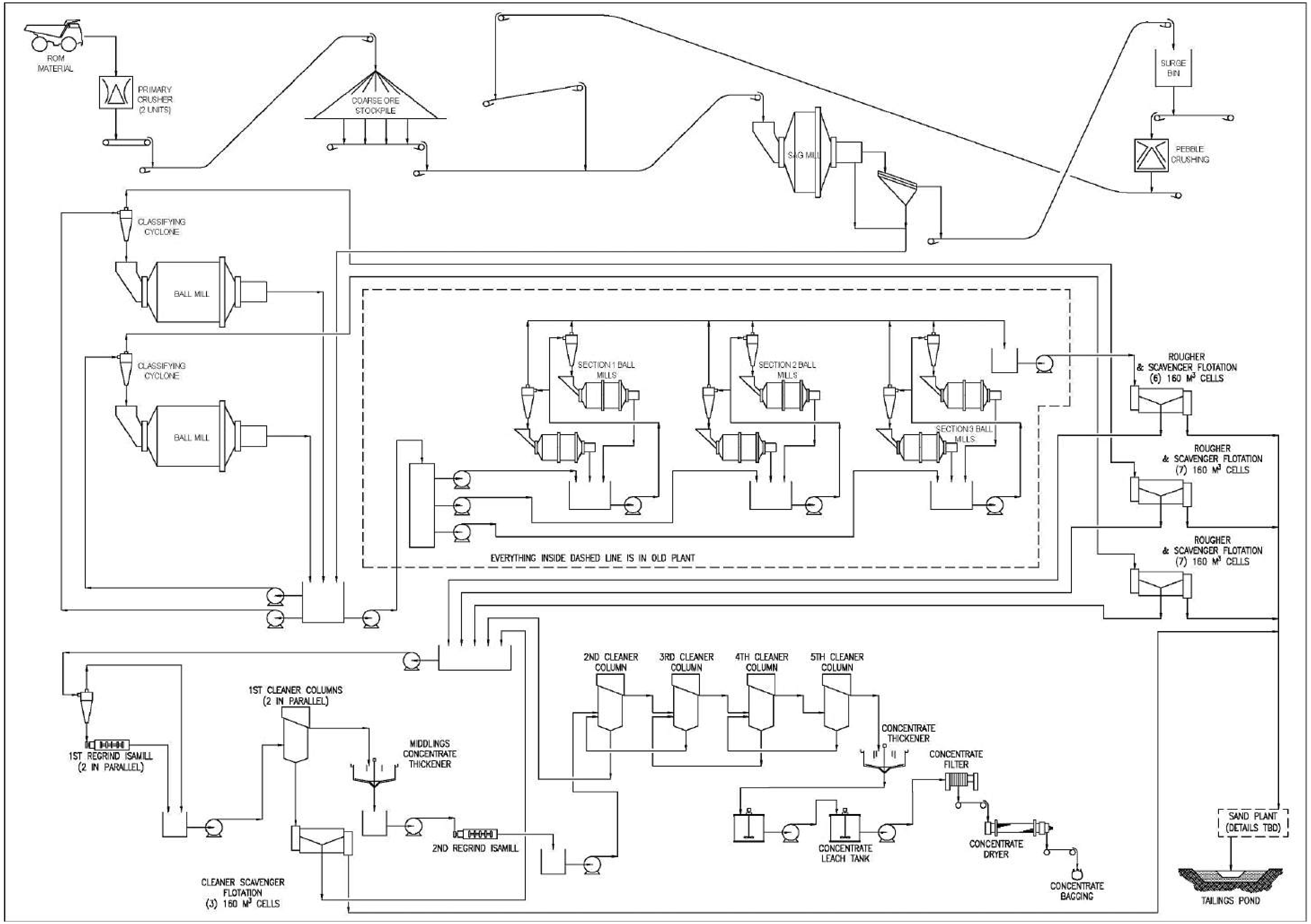


Figure 1.1 Endako Processing Flowsheet

***Source:** AMPL & Hatch*

1.8 INFRASTRUCTURE

Existing infrastructure includes:

1. Access roads, BC power grid power supply to site, nearby railway line and fresh water supply network;
2. Primary crushing plant (to be moved);
3. One new processing plant capable of processing 52,000 tonnes per day;
4. One old decommissioned processing plant capable of processing 30,000 tonnes per day.
5. Tailings management facility with 2 disposal areas TP-1 and TP-3;
6. Reclaim water ponds;
7. Administration, warehouse, change house, laboratory, mine shops and buildings;
8. One reclaimed tailings disposal area TP-2; and
9. Two non-operational roasters.

Refurbishing and upgrading of all facilities, other than the existing roasters that will remain non-operational, will be required and included in the restart plan and cost estimates.

1.9 MANPOWER

The operation would employ a total of approximately 500 hourly and staff personnel.

1.10 ENVIRONMENT AND PERMITTING

The Endako Mine is in compliance with all necessary permits for its status as an operation on care and maintenance. A restart of mining operations will necessitate the reactivation and refreshing of all necessary water and operating permits.

Additionally, an amendment to the existing tailings management facility permit will be required, prior to production. This will allow placement of classified tailings to increase the dam heights, in order to accommodate LOM tailings disposal. It is anticipated that the required permits for a restart of mining operations will be obtained as a matter of course.

1.11 PROJECT SCHEDULE

The Endako Mine restart schedule requires approximately 1.5 to 2 years from the approval of restart to the start of production. During the first year, detailed engineering on the open pit mining, processing plants reconfiguration and upgrades, support facilities refurbishment and upgrades, preparation and awarding of contractor and supplier major contracts will be completed. Construction and start of commissioning will require an additional year.

1.12 CAPITAL EXPENDITURES

Pre-production capital expenditures for the Project Base Case are estimated to total $493.7 million. The total capital expenditure includes contingencies from 20% to 30%. The breakdown of capital expenditures is presented in Table 1.2, below.

TABLE 1.2 PRE-PRODUCTION CAPITAL EXPENDITURES – ESTIMATES	
Component	**Total Expenditures ($ million)**
Mine	$35.1
Equipment Lease Deposit and Purchases	$23.3
Processing Plant	$89.2
Tailings Management Facilities	$150.1
Surface Infrastructure	$13.2
Non- Mining Mobile Equipment	$5.0
Water Management	$31.5
Water Treatment	$52.7
Owner's Costs	$10.0
Contingency	$83.7
Total	**$493.7**

In addition to the capital expenditures, working capital of $57.2 million, based on 3 months of operating costs, has been estimated.

LOM sustaining capital requirements of $3.2 million are estimated. This comprises upgrades to the processing plant and surface infrastructure. Mining equipment is to be leased to the owner for the LOM. The tailings management facility dam raising costs are included in operating costs as the tailings will be used to increase dam heights as part of the tailings facility management plan.

1.13 OPERATING COSTS

The estimated total average operating cost (excluding smelting and refining) is approximately $11.84 per tonne of potentially economic mineralisation or the equivalent (exchange rate of CA$:US$ = 1.35) of US$11.61 per pound of molybdenum. Table 1.3, below, presents a summary of the LOM average operating costs for each department on a cost per tonne of potentially economic mineralisation basis.

TABLE 1.3 PROJECT OPERATING COSTS SUMMARY	
Component	**Cost/tonne ($)**
Mining	$5.45
Processing and Tailings	$5.53
Surface Department, Environmental, and G&A	$0.86
Total Operating Cost per Tonne Ore	**$11.84**
Total Operating Cost per Pound of Molybdenum	**US$11.61**

1.14 FINANCIAL RESULTS

The financial returns (Table 1.4, below) from the potentially economic mineralisation are presented for the Base Case of expected parameters and costs at a molybdenum price of US$47.79 per kilogram (US$22.50 per pound) of molybdenum oxide and an exchange rate of CA$1.00 = US$0.74.

TABLE 1.4
BASE CASE FINANCIAL RETURNS

	Pre-Tax	After-Tax
Pre-production CAPEX ($ millions)	$493.7	$493.7
Undiscounted Net Revenue ($ millions)	$5,854	$5,854
Undiscounted Total Cash Flow ($ millions)	$2,087	$1,478
NPV (5%) – millions	$1,405	$996
NPV (8%) – millions	$1,116	$790
IRR	46%	40%
Payback Period	2.2 years	2.2 years

The mine is forecast to produce approximately 9.1 million kilograms (20.1 million pounds) per year of molybdenum metal in concentrate. LOM total molybdenum metal in concentrate production is 92.7 million kilograms (204.4 million pounds).

1.15 SENSITIVITY ANALYSIS

Sensitivity analysis was performed for molybdenum price, capital expenditures, operating costs, mined grades and exchange rate for ranges up to ±20%. The Project is sensitive to changes in metals prices and reasonably sensitive to changes in all the other variables.

The sensitivity analysis results are shown in Table 1.5 and Table 1.6 and Figure 1.2 and Figure 1.3. Financial results are most sensitive to grade, exchange rate and molybdenum price changes and least sensitive to capital expenditures and operating costs.

TABLE 1.5
SENSITIVITY ANALYSIS FOR AFTER-TAX NPV

Parameter	After-Tax NPV 5% ($ million)								
	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Mined Grade	114	313	473	635	790	945	1,100	1,254	1,409
Molybdenum Price	-302	-31	265	526	790	1,061	1,344	1,641	1,948
Operating Costs	1,071	1,001	931	862	790	719	649	573	502
Capital Costs	865	846	828	808	790	771	753	734	716
US$:CA$ Exchange Rate	273	407	536	666	790	915	1,038	1,161	1,284

Table 1.6 Sensitivity Analysis for After-Tax IRR									
Parameter	**After-Tax IRR (%)**								
	-20%	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**
Mined Grade	14	21	28	34	40	46	52	58	64
Molybdenum Price	-7	7	19	30	40	51	62	73	84
Operating Costs	52	49	46	43	40	37	35	31	28
Capital Costs	50	48	45	43	40	38	37	35	33
US$:CA$ Exchange Rate	19	25	30	36	40	45	50	55	59

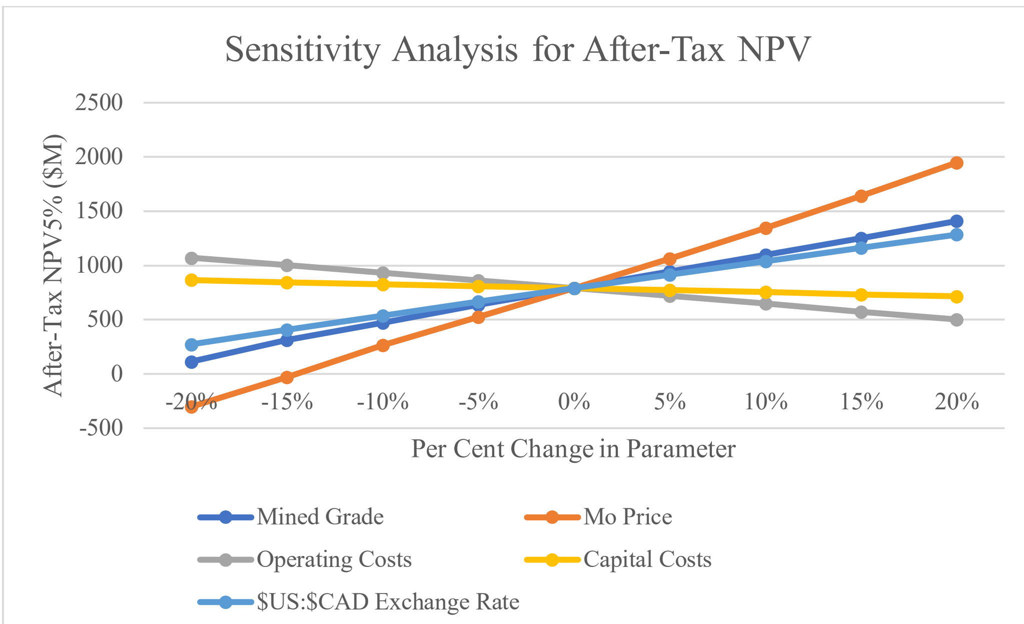


Figure 1.2 Graph of Net Present Value (NPV) at 5% Discount Rate

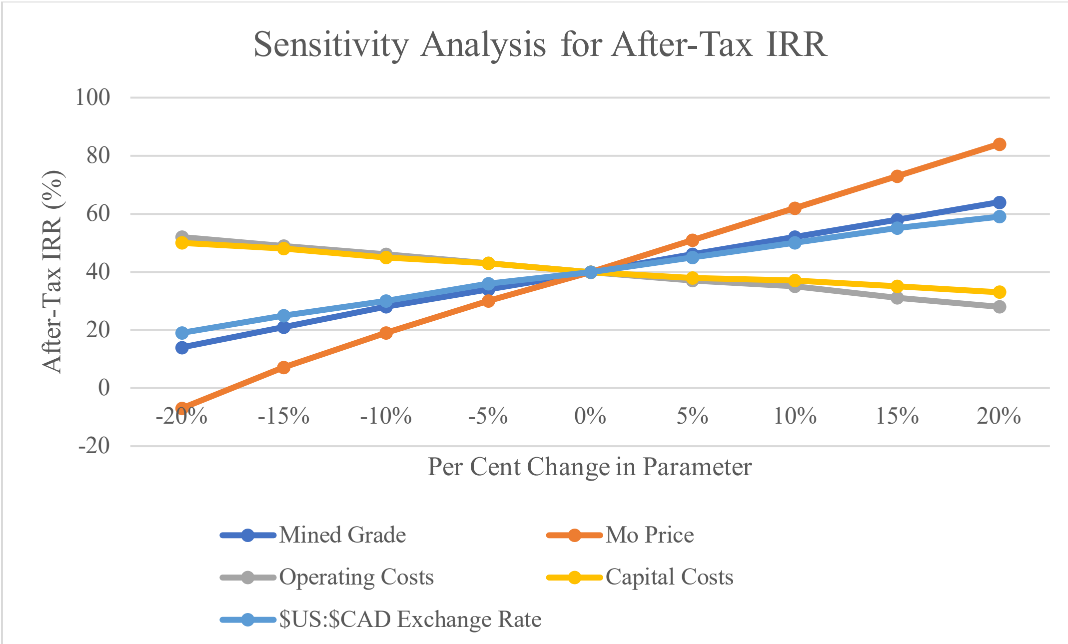


Figure 1.3 Graph of IRR Sensitivity Analysis

1.16 POTENTIAL PROJECT ENHANCEMENTS

Preliminary assessment of shovel/excavator whole bucket ore and waste differentiation has been investigated. This would potentially facilitate better in-pit grade control and minimize waste rock sent to the processing plant for beneficiation. In-bucket sensors determine if a loaded bucket of material is ore or waste, for placing in the appropriate truck for transport to the primary crusher or waste storage areas.

Ore Particle Sorting technology, whereby rock exiting the primary crushers can be screened for an optimum size, should be further investigated. There is currently a 5-tonne representative sample of rock from the Endako pit undergoing testing at a reputable vendor and manufacturer of Ore Particle Sorting equipment.

Production and sale of aggregate from existing and future waste rock production may have economic potential. This could provide pre-production cashflow and added project cashflow over the LOM. Initial petrographic studies indicate that the aggregate is suitable for fill material and may be suitable for concrete. Next steps are to undertake Alkali Aggregate Reactivity testing (using CSA standards to confirm aggregate suitability for concrete) and complete an aggregate marketing study.

1.17 CONCLUSIONS

This PEA examines the viability of restarting open pit mining and flotation processing of the Mineral Resources reported in this PEA Technical Report. The results from this PEA indicate the Endako Project can generate positive economic returns.

The Resources of the Endako Mine comprise of Measured, Indicated and Inferred Mineral Resources. It should be noted that the Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. Therefore, there is no guarantee that the economic projections contained in this PEA would be realised.

Using a cut-off grade of 0.04% MoS_2, there is a Measured and Indicated Resource of 336 million tonnes at 0.072% MoS_2 and an Inferred Resource of an additional 60 million tonnes at 0.054% MoS_2 available for mining. This PEA has identified a diluted potentially mineable resource of 273 million tonnes at an average grade of 0.075% MoS_2 (0.045% Mo) after dilution and mining losses.

The restart plan includes mining and processing 75,000 tonnes per day of potentially economic mineralisation with refurbishing and reconfiguration of the 2 existing processing plants on site (the New Plant on care and maintenance and the Old Plant mothballed).

The existing tailings management facility consisting of TP-1 and TP-3 will be re-commissioned and merged for tailings storage using the existing method of water cover of disposed tailings. The existing tailings dams will be increased in height, using hydrocycloned tailings course fractions, to accommodate the planned potentially economic mineralisation mining and processing rate.

The support infrastructure and facilities are all existing and requiring some upgrading and refurbishing.

The overall expected economic results indicate the Project is economic, generating undiscounted cashflows (over the 10-year LOM) of approximately $2.1 billion and $1.5 billion on a before and after-tax basis, respectively.

At the forecast molybdenum metal price (US$49.73 per kilogram or US$22.50 per pound of molybdenum), the Project's after-tax $NPV_{8\%}$ is estimated to be approximately $790 million. The after-tax Internal Rate of Return (**IRR**) is estimated to be 40%, with payback of approximately 2.2 years from start of production.

Sensitivity analysis shows that the Project economics are most sensitive to mined grade, exchange rate and metal price. The Project is least sensitive to capital expenditures.

Based on the study results, AMPL concludes that the Endako Mine restart provides positive returns based on the parameters and metal prices used in this study.

The Endako Mine restart plan should immediately progress to a Feasibility Study to support financing of the restart of operations.

1.18 RECOMMENDATIONS

Based on this PEA, Endako Mine restart recommendations are:

- Complete a Feasibility Study for mine restart using a mining and processing rate of approximately 75,000 tonnes per day (27 million tonnes per year) of potentially economic mineralisation. A new block model using metric units will be required. (Estimated cost is $4-6 million.)
- Develop an updated or new detailed water management model and Tailings Management Facility (**TMF**) design, which would include hydrology and TMF leachate seepage data and forecasts. (Estimated cost is $500,000.)
- Initiate renewal of permits to encompass the proposed operations immediately upon the initiation of further studies.

- Engage with area Indigenous rightsholders and the Water Quality Working Group on the proposed plans early in the Feasibility Study process.

2.0 INTRODUCTION

2.1 TERMS OF REFERENCE

The Endako Mine Restart, British Columbia, Canada, PEA was prepared for Moon River of Toronto, Canada, by AMPL, Canada.

Moon River, in May 2024, indirectly acquired, through the acquisition of Sojitz Moly Resources Inc. (“**SMR”**), a wholly-owned subsidiary of Sojitz, a 25% participating interest in the Endako molybdenum mine complex. SMR was the holder of a 25% participating interest in the Endako Mine pursuant to an exploration, development and mine operating agreement dated as of June 12, 1997 (the “**Endako** JVA”) entered into between SMR and Thompson Creek Mining Ltd. (now Thompson Creek Metals Company Inc.) (“**TCM**”), a subsidiary of Centerra.

This PEA assesses the potential economic viability of the Project. The cost estimates fall within the guidance on accuracy for PEAs (±40%). The report is prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and in accordance with the requirements of Form 43-101 F1. The disclosure is based on reliable information, the professional opinions of independent Qualified Persons and uses industry best practices and standardised terms.

This PEA provides a plan and the expenditures estimates to restart and operate the Endako Mine and provides guidance to Moon River on how best to progress the Endako Mine Joint Venture to restart operations, using the existing mineral resources.

As of the date of this Report, Moon River Ltd. is a Canadian junior exploration and development company that has its common shares listed on the TSX Venture Exchange (TSXV) with a corporate office at:

Moon River Moly Ltd.
100 King Street West, Suite 7010
Toronto, Ontario M5X 1B1
CANADA
E-mail: pparisotto@moonrivermoly.com

This Report is considered effective as of November 21, 2025 with a filing date of January 5, 2026.

AMPL’s Qualified Persons are responsible for the sections of this report, identified in their “Certificates of Qualified Persons” submitted with this report to the Canadian Securities Administrators. AMPL has relied on and believes there to be a reasonable basis to rely on the following experts who have contributed the information stated in this report, as noted below:

- Mr. Brian LeBlanc, P.Eng, President and Senior Engineer, AMPL.
- Mr. Finley Bakker, P.Geo, Consulting Resource Geologist and Geology to AMPL.
- Mr. Cameron Lilly, P.Eng., Consulting Metallurgist at Concentrator Support Ltd.
- Mr. Will Coverdale, P.E., Coverdale Consulting Ltd.
- Mr. Scott C. Elfen, P.E., P.Eng., Global Lead, Geotechnical and Civil Services, Ausenco Engineering Canada ULC
- Mr. Jonathan Cooper, P.Eng., Water Resources Engineer, Ausenco Sustainability ULC

2.2 SOURCES OF INFORMATION

This study also relies on information provided by Centerra regarding property ownership and mineral tenure (Section 4.0).

Centerra also supplied the geological database and Quality Assurance/Quality Control (QA/QC) data, which is stored at the mine in written paper/hard copy form. This material was reviewed by Mr. Finley Bakker, P. Geo. and verified as much as possible during and subsequent to site visits.

The geological database and QA/QC database is based on electronic and written information and data stored at the mine and Centerra Offices.

Capital assessments of the existing mine facilities relied on detailed assessment reports prepared by Hatch and WSP-Golder in 2022 and 2023 for Centerra for an internal Feasibility Study.

AMPL has assumed, and relied on the fact, that all the information and existing technical documents listed in the References section (Section 27.0) of this Report are accurate and complete, in all material aspects. While AMPL carefully reviewed all the available information provided, AMPL cannot guarantee its accuracy and completeness.

Although copies of the tenure documents, operating licenses, permits and work contracts were reviewed, an independent verification of land title and tenure was not performed. AMPL did not independently verify the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has relied on the client’s solicitor to have conducted the proper legal due diligence. Information on tenure and permits was obtained from Centerra.

Other sources of information and data are provided in Section 3.0.

2.3 SITE VISIT

A site visit was undertaken by Mr. Brian LeBlanc, P.Eng. and Mr. Finley Bakker, P.Geo. Qualified Persons under the terms of NI 43-101 and both with AMPL from April 9-10, 2025. They were accompanied by Mr. Jim Woolsey (Mine Manager, Endako).

A further site visit was undertaken by Mr. Cameron Lily, P.Eng and Mr. Finley Bakker, P.Geo, September 10-11, 2025. They were accompanied by Mr. Mike Pond (retired Chief Geologist) and Mr. Jim Woolsey (Mine Manager, Endako).

On September 15, 2025, a site visit was undertaken by Mr. Scott C Elfen, P.E., Ausenco. During his time at the Project site, Mr. Elfen met Mr. Jim Woolsey from Centerra to take a tour of the Endako Mine’s tailings facilities and to review the facilities’ history and current conditions.

The objective of the field visits was to inspect the site and existing facilities and determine existing and required mine, processing and infrastructure requirements for the Project and gather data from existing reports.

2.4 UNITS OF MEASURE

Unless otherwise stated:

- All measurements used are metric, except where otherwise stated.
- All costs are provided in Canadian Dollars unless otherwise stated.
- Maps are either in UTM coordinates or in the latitude/longitude system.

2.5 GLOSSARY AND ABBREVIATIONS OF TERMS

Abbreviation	**Meaning**
3D	three-dimensional
°C	degrees Celsius
C$ and CA$	currency of Canada
AAS	Atomic Absorption Spectroscopy
AMPL	A-Z Mining Professionals Ltd.
ARD	acid rock drainage
BCEAA	British Columbia Environmental Assessment Act
BCUC	British Columbia Utilities Commission
$CaWO_4$	scheelite
CDA	Canadian Dam Association
cm	centimetre
CO_2	carbon dioxide
DDH	diamond drill hole
E	East
EAO	Environmental Assessment Office
EM	electromagnetic
EMA	Environmental Management Act
ENV	Ministry of Environment and Climate Change Strategy
EPCM	Engineering, Procurement, and Construction Management
EPIC	Environmental Assessment Offices Project Information Centre
ESSFmc	Engelmann Spruce Subalpine Fir
°F	degrees Fahrenheit
FS	Feasibility Study
FTSF	Filtered Tailings Storage Facility
ft^3/tonne	cubic feet per tonne
g	gram
g/t	grams per tonne
ha	hectare
HDPE	High Density Polyethylene
HP	horsepower
IAA	Impact Assessment Act
IBA	Impact Benefit Agreement
ICHmc1	Interior Cedar Hemlock
ICP-ES	Inductively Coupled Plasma-Emission Spectrometry
IP	induced polarisation
IRR	Internal Rate of Return

kg	kilogram
km	kilometre
kmph	kilometres per hour
kW	kilowatts
kV	kilo volt
kVA	kilo volt-amperes
LLDP	Linear Low Density Polyethylene
LME	London Metal Exchange
LOM	Life-of-Mine
m	metre
m^2	square metre
m^3	cubic metre
m.a.s.l.	metres above sea level
MCC	master control centre
MCM	thousands of circular mils
mm	millimetre
Mo	molybdenum
MOE	Ministry of Environment
MoO_2	molybdenum oxide
MoO_3	molybdenum trioxide
MoS_2	molybdenum disulphide
MS3D	MineSight 3D
Mt	million tonnes
MVA	megavolt-amps
N	North
NPV	Net Present Value
NSR	Net Smelter Return
NPVS	NPV Scheduler
opt	ounces per ton
PAG	potentially acid generating
PDC	process design criteria
P.Geo	Professional Geoscientist
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
PMP	Probable Maximum Precipitation
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
$SBSmc^2$	Sub-Boreal Spruce
SG	specific gravity
SM^2	Special Management 2
t	tonne (metric)
t/a	tonnes per annum
t/m^3	tonnes per cubic metre
t/d	tonnes per day
TCMC	Thompson Creek Metals Company
TDEM	time domain electromagnetic
TMF	tailings management facility
THO	Trans-Hudson Orogen
TP	Tailings Pond

US$	currency of the United States of America
USA	United States of America
UTM	Universal Transverse Mercator
V	volt
VMS	Volcanogenic Massive Sulphide
WRSF	Waste Rock Storage Facility
WTP	water treatment plant

3.0 RELIANCE ON OTHER EXPERTS

This PEA also relies on the assessments and estimates prepared by the following consulting companies as follows:

- Endako Restart Project Engineering Report, Hatch, 2022;
- Basis of Estimate – Endako Restart Project, Hatch, 2022;
- Feasibility Design of Tailings Ponds 1 and 3 for Life of Mine Plan – Endako Mine, Golder Associates Ltd., 2022;
- Feasibility Mine Planning Study for Endako Mine, WSP Golder, May 2022 and
- Water Treatment Design Basis for the Endako Mine Restart, SRK Consulting (Canada) Inc., 2022.

AMPL used and/or relied on the services of the following consultants and firms:

- Ms. Michelle Tanguay, President, 2Tango Environmental Services, Section 20.0.

The Qualified Persons have taken reasonable measures to confirm information provided by others (including the listed consultants) and take responsibility for the information.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Endako Mine Project is an open pit molybdenum mine and processing plant located approximately 190 km northwest of Prince George, British Columbia, Canada. The latitude and longitude of the Project site are 54° 02′ north latitude and 125° 07′ west longitude or 5990212 metres (m) north and 362020 m east, UTM Zone 10, NAD 83. The location of the Endako mining property is shown in Figure 4.1, Figure 4.2 and Figure 4.3, below.

Figure 4.1. Location Map
Source: *WGM, January 2018 – modified from Davenport map*

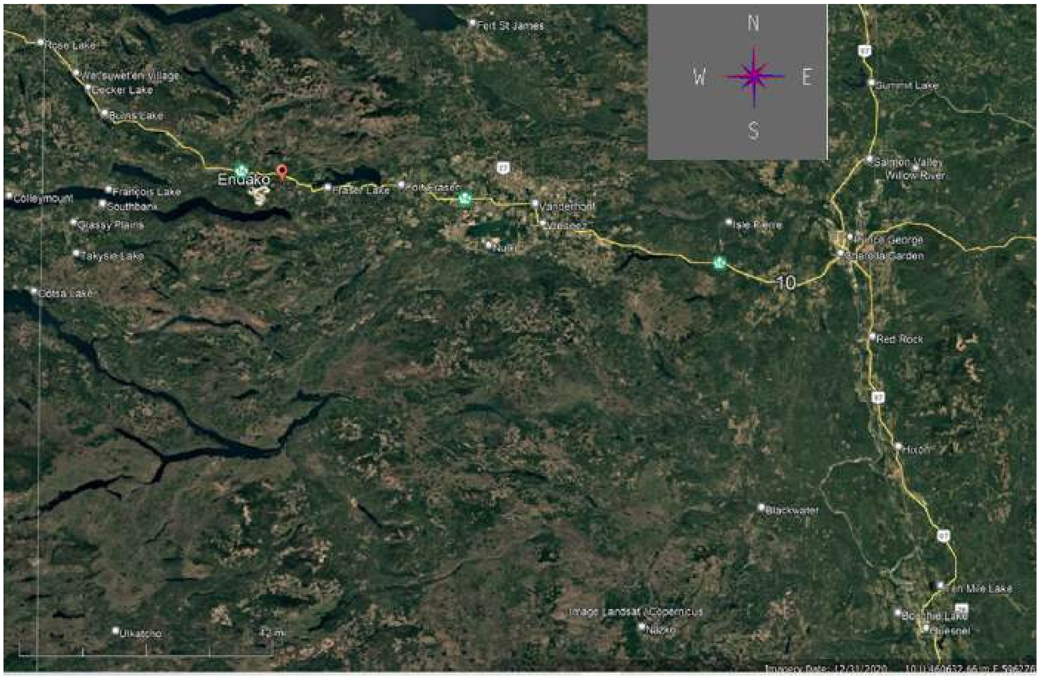


Figure 4.2 Location of Endako
***Source:** AMPL, 2025*

Figure 4.3 Location of Endako Site
***Source:** AMPL 2025*

The Endako Mine also has a roaster complex, which will not be recommissioned in the future.

The past operating mine was placed in 'Care and Maintenance' on July 01, 2015. At the time of the production suspension, the Endako Mine was operating at a processing rate of approximately 52,000 tonnes per day.

Moon River, in May 2024, indirectly acquired, through the acquisition of SMR, a wholly-owned subsidiary of Sojitz, a 25% participating interest in the Endako molybdenum mine complex. SMR is the holder of a 25% participating interest in the Endako Mine pursuant to an exploration, development and mine operating agreement dated as of June 12, 1997 (JVA) entered into between SMR and Thompson Creek Mining Ltd., now TCM, a subsidiary of Centerra.

4.1 MINERAL TENURE AND AGREEMENTS

The Property consists of 34 legacy and converted legacy claims and 26 mineral leases covering an area of approximately 10905.2 ha (Figure 4.4, below). As of August 1, 2025, information extracted from the British Columbia mineral title online database indicated all claims and leases are in good standing (Table 4.1, below). The mine property boundaries have all been surveyed.

Located or ground-staked mineral claims are termed "legacy" claims. These claims can be converted to cell claims to provide a secure title, eliminate mapping issues, consolidate smaller titles into larger ones and reduce overlap.

Mining leases are issued according to a survey plan and for a specific term. There are no work requirements on a lease, but the term of a lease will only be renewed if the lease is required for mining activity.

AMPL validated the claim and lease data provided by Endako against the information provided by the British Columbia MTO database and the information provided by Endako was found to be correct. No other validation was performed by AMPL regarding the mineral tenure.

Table 4.1, below, lists the mineral claims comprising the Project. All mineral claims are indicated to be in good standing until the earliest date of May 2026.

4.2 PERMITS AND ENVIRONMENTAL LIABILITIES

The Endako Mine holds all necessary permits scaled for the current operation in care and maintenance. The Mines Act Permit and Water Management Act permits will need to be amended to restart operation. The amendments will include new discharge criteria.

As the area of disturbance for the restart of operations will not be 50% or greater of the area already previously permitted, an environmental assessment will not be required.

AMPL is not aware of any environmental liabilities, other than closure requirements associated with the Property at this time.

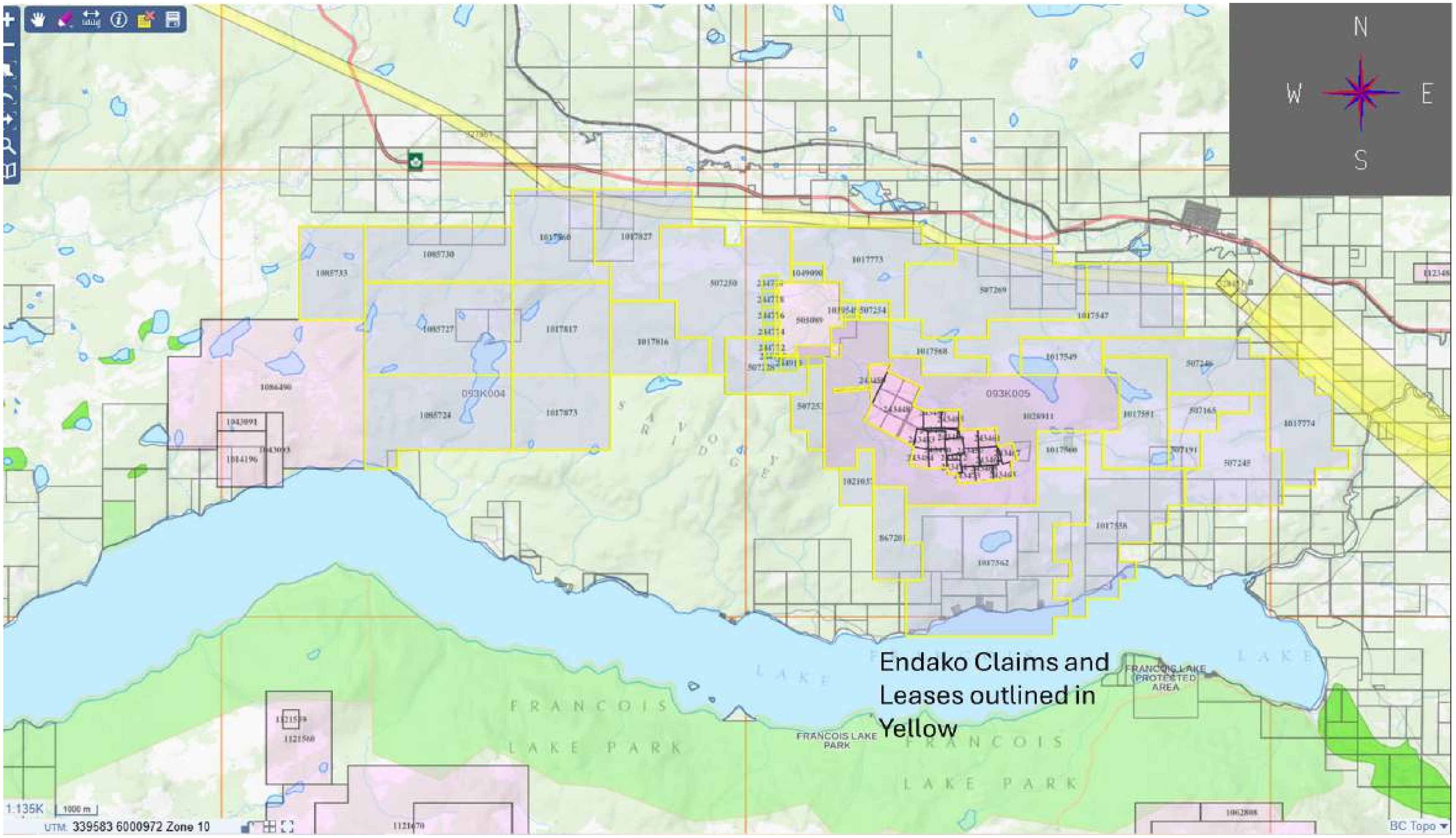


Figure 4.4 ***Mining Lease and Claims***

Source: *British Columbia Mineral Titles Online, 2025*

TABLE 4.1
LIST OF CLAIMS AND LEASES

Title Number	Claim Name	Owner	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Status	Area (ha)
243448		290004 (100%)	Mineral	Lease	093K005	1977/MAY/06	2026/MAY/06	GOOD	164.53
243450		290004 (100%)	Mineral	Lease	093K005	1979/SEP/06	2026/SEP/06	GOOD	36.92
243457		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	19.55
243458		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	18.52
243459		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	19.75
243460		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	20.9
243461		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	20.81
243462		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	0.73
243463		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	18.19
243464		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	18.84
243465		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	2.05
243466		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	7.12
243467		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	16.78
243468		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	17.26
243469		290004 (100%)	Mineral	Lease	093K005	1964/SEP/23	2026/SEP/23	GOOD	0.2
243470		290004 (100%)	Mineral	Lease	093K005	1967/JAN/05	2027/JAN/05	GOOD	20.19
243471		290004 (100%)	Mineral	Lease	093K005	1967/JAN/05	2027/JAN/05	GOOD	16.25
243472		290004 (100%)	Mineral	Lease	093K005	1967/JAN/05	2027/JAN/05	GOOD	0.09
243473		290004 (100%)	Mineral	Lease	093K005	1967/JAN/05	2027/JAN/05	GOOD	16.3
243474		290004 (100%)	Mineral	Lease	093K005	1967/JAN/05	2027/JAN/05	GOOD	2.06
243482		290004 (100%)	Mineral	Lease	093K005	1971/JAN/29	2027/JAN/29	GOOD	2.72
243483		290004 (100%)	Mineral	Lease	093K005	1971/JAN/29	2027/JAN/29	GOOD	15.08
243484		290004 (100%)	Mineral	Lease	093K005	1971/JAN/29	2027/JAN/29	GOOD	19.96
243485		290004 (100%)	Mineral	Lease	093K005	1971/JAN/29	2027/JAN/29	GOOD	20.85
243486		290004 (100%)	Mineral	Lease	093K005	1971/JAN/29	2027/JAN/29	GOOD	20.7
244772	SAM 18	290004 (100%)	Mineral	Claim	093K005	1969/APR/17	2029/DEC/11	GOOD	25
244774	SAM 20	290004 (100%)	Mineral	Claim	093K005	1969/APR/17	2029/DEC/11	GOOD	25
244776	SAM 22	290004 (100%)	Mineral	Claim	093K005	1969/APR/17	2029/DEC/11	GOOD	25
244778	SAM 24	290004 (100%)	Mineral	Claim	093K005	1969/APR/17	2029/DEC/11	GOOD	25
244780	SAM 26	290004 (100%)	Mineral	Claim	093K005	1969/APR/17	2029/DEC/11	GOOD	25
244913	SAM 80	290004 (100%)	Mineral	Claim	093K005	1969/SEP/12	2029/DEC/11	GOOD	25

TABLE 4.1
LIST OF CLAIMS AND LEASES

Title Number	Claim Name	Owner	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Status	Area (ha)
244915	SAM 82	290004 (100%)	Mineral	Claim	093K005	1969/SEP/12	2029/DEC/11	GOOD	25
507165		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2029/DEC/11	GOOD	151.905
507191		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2029/DEC/11	GOOD	75.968
507228		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2029/DEC/11	GOOD	246.781
507245		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2029/DEC/11	GOOD	474.835
507246		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2029/DEC/11	GOOD	398.653
507250		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2029/DEC/11	GOOD	834.877
507253		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2029/DEC/11	GOOD	132.91
507254		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2031/FEB/15	GOOD	37.956
507269		290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2029/DEC/11	GOOD	815.973
1017547	Kendo 02	290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2026/FEB/27	GOOD	664.2547
1017549	Kendo 4	290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2026/FEB/27	GOOD	189.8285
1017551	Kendo 06	290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2026/FEB/27	GOOD	398.759
1017558	Kendo 08	290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2026/FEB/27	GOOD	721.9889
1017560	Kendo 10	290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2026/FEB/27	GOOD	113.953
1017562	Kendo 12	290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2026/FEB/27	GOOD	1,349.265
1017568	Kendo 15	290004 (100%)	Mineral	Claim	093K	2005/FEB/15	2026/FEB/27	GOOD	189.823
1017773	KENDO 20	290004 (100%)	Mineral	Claim	093K	2013/MAR/14	2026/FEB/27	GOOD	531.2414
1017774	KENDO 21	290004 (100%)	Mineral	Claim	093K	2013/MAR/14	2029/DEC/11	GOOD	436.7476
1017816	KENDO 22	290004 (100%)	Mineral	Claim	093K	2013/MAR/15	2029/DEC/11	GOOD	379.6087
1017817	KENDO 23	290004 (100%)	Mineral	Claim	093K	2013/MAR/15	2029/DEC/11	GOOD	569.3725
1017827	KENDO 24	290004 (100%)	Mineral	Claim	093K	2013/MAR/16	2029/DEC/11	GOOD	493.2187
1017860	KENDO 25	290004 (100%)	Mineral	Claim	093K	2013/MAR/17	2029/DEC/11	GOOD	493.2194
1017873	KENDO 25	290004 (100%)	Mineral	Claim	093K	2013/MAR/18	2029/DEC/11	GOOD	455.7034
1039549	ENDAKO1	290004 (100%)	Mineral	Claim	093K	2015/OCT/26	2026/OCT/26	GOOD	18.9781
1049090		290004 (100%)	Mineral	Claim	093K	2017/JAN/11	2026/FEB/27	GOOD	37.9471
TOTAL									**10,905.12**

Source: *British Columbia Mineral Titles on Line 2025*
Source: *AMPL, 2025*

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

5.1 ACCESSIBILITY

The Endako Mine is located in the Bulkley-Nechako region of central British Columbia, approximately 190 km west of Prince George and about 400 km east of Prince Rupert, British Columbia. The nearest town is Fraser Lake, about 15 km east of the Endako Mine. The Endako Mine is located about 10 km south of Highway 16 (the Yellowhead Highway) on a paved road. The village of Endako is on Highway 16 at the junction with the Mine Road. The highways to Fraser Lake and Prince George are paved, high quality roads that are part of the Trans-Canada Highway system.

5.2 CLIMATE

The average temperature ranges from -9.5°C in January to 16.3°C in July. The average snowfall from the beginning of November to the end of March is 30.6 centimetres (cm) with an on-ground accumulation averaging 6 cm in November, 33 cm in January and 15 cm in March. The frost-free period is from June to September. During the summer months, the average rainfall from the beginning of April to the end of October is 41 millimetres (mm). The Endako Mine can operate year around.

5.3 PHYSIOGRAPHY

The Endako Mine is located on the Nechako Plateau near the geographical center of British Columbia. The area is characterised by broad valleys, flat-topped hills and generally gently rolling terrain. Glaciation moved across the area from the west leaving a distinct east-west grain. Elevations range from 670 metres (m) at Endako village to 1,070 m at the crest of the Endako pit.

The Endako Mine area is generally forested with pine and spruce with areas of open grassland. The uplands are well drained with a few marshes and lakes, while the valleys are bottomed by narrow lakes, such as the Fraser Lake and Francois Lake.

5.4 LOCAL RESOURCES

Prince George is the largest service center in Northern British Columbia. Fraser Lake is 20 km to the northeast of the Endako Mine and is the nearest significant community to the Endako Mine. A network of mine roads provides excellent access to most parts of the Property. A railway line owned by Canadian National (CN) parallels the Yellowhead Highway north of the Mine. Electrical power is provided by an 8.5 km long 69 kilovolt (kV) power line from the village of Endako. The power line is owned by B.C. Hydro.

Fresh water for operations is pumped from the nearby Francois Lake.

5.5 EXISTING FACILITIES AND INFRASTRUCTURE

The present existing facilities and infrastructure includes:

- Three open pit mines Endako, Denak West and Denak East;
- 52,000 tonnes per day processing plant (constructed in 2011/2012), 2 tailings management areas TP-1 and TP-3;
- The original 30,000 tonnes per day mill, which has been mothballed.
- One reclaimed tailings area – TP-2;
- Maintenance facilities and warehouse;
- Assay laboratory;
- Technical and administration offices; and
- Access roads, grid connected power supply, nearby railway line, and other support services for the operation.

All facilities and services are in place though refurbishing, improvements and upgrades are required prior to restart of operations. The existing facilities and proposed changes are described in each relevant section of this report.

6.0 HISTORY

6.1 HISTORIC EXPLORATION ACTIVITIES

The Endako deposit was originally discovered in 1927. Minor underground exploration work took place in the subsequent years. Exploration continued on and off through 1959. Due to the leached nature of the mineralisation, extensive overburden, low-grades and lack of precious metals, the Property was dropped in 1958.

In 1962, R&P Metals Corporation Ltd. began a diamond drilling program to evaluate the discovery and based on the exploration results, incorporated a company named Endako Mines Ltd. (Endako). Canadian Exploration Limited, a wholly-owned subsidiary of Placer Development Ltd. (Placer) then entered into an option agreement with Endako in August 1962 and continued exploration on the Property with diamond drilling and bulk sampling.

In March 1964, Placer decided to place the Property into production. Production commenced in June 1965 at a plant capacity of approximately 9,070 tonnes per day. Endako merged with Placer in 1971. Expansions during 1967 increased production to 24,500 tonnes per day and improvements during 1980 increased the concentrator capacity once more to 28,000 tonnes per day to 30,000 tonnes per day.

The Endako Mine and concentrator were closed from 1982 to 1986 due to poor demand for molybdenum. The roaster continued to operate, processing molybdenum concentrates from other operations on a toll basis. Mine and concentrator production resumed in 1986 at a reduced rate. Production returned to 28,000 tonnes per day by 1989.

TCM (75%) and Nissho Iwai Moly Resources, Inc. (25%) formed the Endako Joint Venture and acquired the Property from Placer in 1997. Nissho Iwai Moly Resources Inc. later changed its name to Sojitz.

Blue Pearl Mining Ltd. acquired TCM in October 2006, and, on May 14, 2007, Blue Pearl Mining Ltd. changed its name to TCM.

Endako was issued a Mines Act permit amendment in early March 2012 allowing the increase in production. The expansion amalgamated Endako Mine’s three pits and resulted in a major upgrade to Endako Mine’s 42-year-old mill and created a new facility that nearly doubled the processing capacity from 28,000 tonnes per day to 55,000 tonnes per day.

The Endako Mine was put on “temporary suspension” in late December 2014 due to a weak market for molybdenum. The price of molybdenum, in early December 2014, struck a low of US$8.95 per pound to US$9.05 per pound. When the expanded mill was opened in June 2012, the price stood at US$13 per pound, which was down from US$17 per pound the year before.

On July 5, 2016, Centerra and TCM announced they had entered into a definitive arrangement whereby Centerra was to acquire all the issued and outstanding common shares of TCM. The plan of arrangement was completed on October 20, 2016. From that date until May 2024, the Endako Mine was a joint venture between Centerra (75% interest) and Sojitz (25% interest). In May 2024, Moon River indirectly acquired, through the acquisition of SMR, a wholly-owned subsidiary of Sojitz, a 25% participating interest in the Endako molybdenum mine complex. SMR is the holder of a 25% participating interest in the Endako Mine pursuant to the Endako JVA.

Exploration has been ongoing from the mid-sixties to the present, including geochemical sampling, diamond drilling and percussion drilling. Recent work conducted by Placer, since 1989, includes 14 diamond drill holes in 1989, 22 in 1992, 44 in 1993 and 19 in 1994.

Exploration resumed in 2001. Since that year, Endako carried out yearly exploration programs, except in 2005, 2009, 2012 and 2013. These programs consisted mainly of diamond drilling with geophysical surveys in 2004 and 2006 and metallurgical testing in 2002. A LiDAR survey was completed in 2013 along with a legal ground survey of the mineral claims and mineral leases.

Endako also engaged the service of Integral Ecological Group in 2013 to design and implement a pilot reclamation assessment program. This work was continued in 2015.

During the 2016 summer field season, a small geochemical orientation survey was conducted on the claim and surrounding area to re-establish molybdenum anomalies from historical surveys completed in the 1970s (Bysouth, G.D., 1970; Kimura, 1970; Nilsson, 1979).

6.2 HISTORICAL RESOURCE ESTIMATES

Endako Mine was an operating mine until it closed in December 2014. Resources and Reserves were continually updated during the Endako Mine life. Two reports were referenced in this report.

- Marek J., September 12, 2011, *NI 43-101 Technical Report, Endako Molybdenum Mine*, Independent Mining Consultants Inc. (IMC).
- Desautels, P., Watts, Griffis and McOuat Limited (WGM), *Internal Technical Report Centerra Gold Inc. – Endako Mine Technical Report NI 43-101* – March 2018.
- *Endako Mine Inferred Resource and Reserve Estimates* – December 2014.

Endako Mine was an operating mine until it closed in December 2014. Resources and Reserves were continually updated during the Mine life. The historical Mineral Resources described in this section are considered the most relevant to this study. IMC used a methodology that is somewhat similar to the current resource estimate described in Section 14.0 of this report, while the Endako Mine Resources and Reserves are the latest figures available when the Mine closed.

The Mineral Resource estimates described in this section are now considered historical in nature. They are provided here for historical context only. AMPL is not treating these historical estimates as current Mineral Resources or Reserves and has not undertaken any independent investigation of the resource estimates; therefore, the resource estimates in Table 6.1 to Table 6.4, below, should not be relied upon. These historical resource estimates are no longer current and have been superseded by the resource estimate described in Section 14.0 of this report.

6.2.1 IMC Resource and Reserve Estimates – June 1, 2011

In 2011, Mr. John Marek, P.E., Independent Mining Consultants Inc. (IMC), issued a *Technical Report and Mineral Resource/Reserve Estimate for the Endako Mine*. The September 12, 2011 report followed guidelines conforming to NI 43-101 Standards of Disclosure for Mineral Projects, and follows the format set out in Form 43-101 F1 for Technical Reports. The Resource estimate was based on all available drill data through October 2010 and selected blast holes data through May 31, 2011. The database consisted of a total of 1,046 diamond drill holes aggregating 516,305 feet (ft) and contained 47,957 intervals assayed for molybdenum disulfide (MoS_2). The drill data was supplemented with 13,748 blast holes.

A probabilistic model was constructed to establish a boundary between material with grade above and below the threshold cut-off of 0.035% MoS_2. The grade model was interpolated with ordinary linear kriging respecting the high-grade/low-grade boundaries as defined by the probabilistic model. Statistical and structural domain boundaries for the Endako pit area were incorporated into the model to adjust the orientation of the search ellipse for kriging.

Model blocks were classified as Measured, Indicated or Inferred based primarily on the kriging standard deviation. Tonnages were estimated using density data supplied by Endako. Resources were reported within a floating cone based on CA$16.50 per pound of molybdenum. The floating cone pit geometry was used to establish that the resource has “reasonable prospects of economic extraction.”

The Mineral Resources in Table 6.1, below, were dated June 1, 2011 and were in addition to the Mineral Reserve.

TABLE 6.1
IMC RESOURCE TABLE – JUNE 1, 2011

Category	Cut-off MoS_2 %	K-tonnes	MoS_2 %	Mo %	Contained Metal Million (lbs)
Measured	0.025	17,212	0.048	0.029	10.9
Indicated	0.025	40,156	0.051	0.031	27.1
Measured + Indicated		**57,368**	**0.05**	**0.03**	**38**
Inferred	0.025	49,342	0.058	0.035	37.8

***Source**: IMC 2011*

The Mineral Resource estimates described in this section are now considered historical in nature. The issuer is not treating the historical estimate as current Mineral Resources. They are provided here for historical context only.

The report stated a mine plan was developed by the mine planning staff at Endako. That plan was reviewed by IMC for inclusion within this technical report. The total of all Proven and Probable category reserves that were planned for mill treatment plus the existing mill stockpile that existed on-site were combined to estimate the Mineral Reserves, as summarised in Table 6.2, below. The final pit geometry designed by the Endako team was checked against floating cones developed on the June block model at a range of metal prices from CA$12.00 per pound to CA$15.00 per pound. A floating cone at a molybdenum price of CA$13.50 was selected for the final pit design.

TABLE 6.2
IMC RESERVE TABLE – JUNE 1, 2011

Material Type	Category	Cut-off MoS_2 %	K-tonnes	Grade MoS_2 %	Grade Mo %	Contained Metal Million (lbs)
Mill Ore + Planned Stockpiles	Proven	0.03	96,881	0.081	0.049	103.7
	Probable	0.03	164,099	0.076	0.046	164.8
Existing Stockpiles	Proven	High-Grade	21,861	0.078	0.047	22.5
	Proven	Low-Grade	9,091	0.057	0.034	6.8
In-Pit Dump Stockpile	Probable		16,747	0.065	0.039	14.4
Total Proven			**127,833**	**0.079**	**0.047**	**133**
Total Probable			**180,846**	**0.075**	**0.045**	**179.2**
Total Proven and Probable			**308,679**	**0.077**	**0.046**	**312.2**

Source*: IMC 2011*

The Mineral Reserve estimates described in this section are now considered historical in nature. The issuer is not treating the historical estimate as current Mineral Resources. They are provided here for historical context only

6.2.2 Endako Mine Inferred Resource and Reserve Estimates – December 2014

The information for this resource/reserve was sourced from a memorandum authored by Mr. Robert Clifford, Director, Mine Engineering. The memo is dated January 21, 2015 and describes the Mineral Reserves and Mineral Resources as of December 31, 2014. A second document, authored by Mr. M. Pond, Chief Geologist and dated January 8, 2015, described the interpolation parameters used to estimate the Resources.

It was stated in the memorandum that the Reserves and Resources have been calculated in accordance with the requirements and/or guidelines adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) and incorporated in the NI 43-101.

The Resource estimate was based on all available drill data through December 2014. The document provided does not specify if the drill data was supplemented with blast holes. The grade model was interpolated with inverse distance to the power of 2 (ID^2) with no domain boundaries.

Model blocks were classified as Measured, Indicated or Inferred based on the search ellipsoid dimension. Tonnages were estimated using density data from Endako. In order to meet the "reasonable prospects of economic extraction," the spot market prices were selected to have a reasonable likelihood to be achieved within a 10-year to 25-year time period. To determine the resultant Mineral Resources with respect to reasonable potential spot market prices, the mining software NPV Scheduler v4.21™ (NPVS) was utilised to generate a constraining shell. The Mineral Resources in Table 6.3, below, were dated December 31, 2014 and were in addition to the Mineral Reserve.

TABLE 6.3 ENDAKO MINE MINERAL RESOURCES – DECEMBER 2014			
Category	**Million (tonnes)**	**Mo %**	**Contained Metal Million (lbs)**
Measured	47.6	0.0469	49.2
Indicated	61.6	0.0466	63.3
Measured + Indicated	**109.2**	**0.0467**	**112.5**
Inferred	2.2	0.0392	N/A

Source: *WGM, 2018*

The Mineral Reserve estimates described in this section are now considered historical in nature. The issuer is not treating the historical estimate as current Mineral Resources. They are provided here for historical context only.

For Mineral Reserve classification, the resource pit shell output from NPVS was brought into the computer aided drafting (CAD) design software MineSight 3D v7.80™ (MS3D) to convert the shell (or cone) into a pit design. This calculation is performed at current spot market prices and all other cost and recovery assumptions used for Mineral Resource determination. MS3D was used to quantify and tabulate, according to cut-off, the quantity and quality of economic reserves by tonnage and average grade by bench, by layback. The tabulated reserves were then scheduled with recoveries and costs applied to determine saleable product and the resultant economic benefit. If the resultant key metrics did not match the input parameters, an iterative process of refining assumptions and running through to design and schedule was undertaken to achieve reserve status. The Mineral Reserves are summarised in Table 6.4, below.

TABLE 6.4 ENDAKO MINE MINERAL RESERVE – DECEMBER 2014				
Material Type	**Category**	**Million (tonnes)**	**Mo%**	**Contained Metal Million (lbs)**
Pit	Proven	8.4	0.0557	10.4
Stockpile	Proven	1.7	0.039	1.4
Pit	Probable	9.3	0.0547	11.3
Stockpile	Probable	14	0.0413	12.7
Total Proven and Probable		**33.4**	**0.0486**	**35.8**

Source: *WGM, 2018*

The Mineral Reserve estimates described in this section are now considered historical in nature. The issuer is not treating the historical estimate as current Mineral Resources. They are provided here for historical context only

A comparison was also made to an unpublished draft report "Technical Report on the Endako Mine British Columbia, Canada," Centerra Gold Inc.NI 43-101 report (Table 6.5, below).

TABLE 6.5
2018 RESOURCE STATEMENT

Endako Resources within $14 per lb Constraining Pit Shell at >0.042% MoS_2 (0.025% Mo) Cut-off

Classification	Tonnes (million)	MoS_2 %	Mo %	Mo (million lbs)
Measured	67.8	0.075	0.045	67.6
Indicated	186.0	0.071	0.043	174.4
Measured + Indicated	253.8	0.072	0.043	242.0
Inferred	78.0	0.067	0.040	69.4

Source: *Endako Internal Report*

The Mineral Reserve estimates described in this section are now considered historical in nature. The issuer is not treating the historical estimate as current Mineral Resources. They are provided here for historical context only

6.3 RESULTS FROM PROPERTY PRODUCTION

Endako Mine operated from 1965 through 2014, except when the Endako Mine was shut down due to poor market conditions between 1983 and 1985. Table 6.6, below, presents the Endako Mine production statistics by year, as provided by Centerra Gold Inc.

TABLE 6.6
MINE PRODUCTION STATISTICS BY YEAR

	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985
Tonnes Ore and Waste Mined (000s)	3,817	6,532	13,620	8,661	11,623	13,199	14,133	9,579	12,213	12,080	19,944	19,881	18,457	23,120	8,511	24,645	22,342	8,655	0	0	0
Strip Ratio	0.88	0.84	1.22	0.45	0.33	0.44	0.72	0.65	0.59	0.77	1.33	1.33	1.03	1.17	0.79	1.22	1.13	2.01	0	0	0
Tonnes Milled (000s)	2,035	3,549	6,148	5,983	8,733	9,233	8,209	4,881	7,661	6,810	8,600	8,519	9,083	10,655	4,767	11,101	10,491	2,880	0	0	0
Head Grade % MoS_2	0.183	0.237	0.212	0.178	0.189	0.182	0.162	0.149	0.146	0.165	0.161	0.163	0.161	0.135	0.129	0.141	0.110	0.091			
Recovery %	81.3	81.6	81.0	85.9	86.0	82.4	81.6	81.2	80.0	81.1	83.0	81.9	78.8	73.5	73.2	77.7	76.9	78.6			
Mo Production (000's kgs)	1,817	4,119	6,334	5,489	8,512	8,308	6,511	3,544	5,366	5,465	6,894	6,823	6,914	6,343	2,701	7,297	5,325	1,236	0	0	0

	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006
Tonnes Ore and Waste Mined (000s)	2,352	7,956	10,154	15,579	20,764	22,744	22,516	18,463	19,916	19,220	21,349	19,319	19,859	7,964	14,752	11,120	11,984	14,935	14,459	17,089	21,283
Strip Ratio	0.6	0.69	0.48	0.68	1.14	1.38	1.32	0.94	0.92	0.82	1.01	1.08	0.99	0.78	0.57	0.42	0.26	0.55	0.68	1.29	1.12
Tonnes Milled (000s)	1,466	4,716	6,845	9,265	9,703	9,543	9,701	9,585	10,385	10,430	10,023	9,377	9,805	8,426	9,396	9,386	9,641	9,706	9,350	9,604	9,700
Head Grade % MoS_2	0.147	0.159	0.134	0.127	0.134	0.137	0.150	0.145	0.133	0.140	0.141	0.142	0.128	0.108	0.111	0.128	0.122	0.111	0.112	0.099	0.115
Recovery %	79.2	83.0	83.9	82.9	83.2	82.9	80.1	79.6	74.8	74.7	77.4	76.5	76.7	76.7	76.2	78.2	75.7	80.0	79.2	77.2	78.4
Mo Production (000's kgs)	1,024	3,735	4,617	5,853	6,491	6,503	6,994	6,637	6,199	6,544	6,563	6,112	5,788	4,191	4,740	5,643	5,326	5,185	4,971	4,380	5,253

	Oct _Dec 2006	2007	2008	2009	2010	2011	2012	(July YTD)	Totals
Tonnes Ore and Waste Mined (000s)	4,416	17,277	20,376	18,251	21,149				**666,258**
Strip Ratio	0.76	0.73	0.53	0.83	0.93				**0.82**
Tonnes Milled (000s)	2,279	9,808	10,768	9,759	10,176	10,652	14,711	8,253	**391,795**
Head Grade % MoS_2	0.099	0.100	0.116	0.098	0.100	0.090	0.070	0.080	**0.136**
Recovery %	75.7	72.8	77.7	78.4	74.5	74.0	62.7	69.4	**78.5**
Mo Production (000s kgs)	1,028	4,275	5,799	4,517	4,553	4,233	3,873	2,747	**242,775**

***Source**: Centerra Gold Inc.*

7.0 GEOLOGICAL SETTING AND MINERALISATION

Earth scientists from several government agencies, eleven universities, and four companies joined forces, both formally and informally, from 1995 to 2000, to study Eocene tectonics in the central Canadian Cordillera of British Columbia. Their research was conducted under the auspice of the Geological Survey of Canada's National Mapping Program (NATMAP) as the Nechako Project, which covered the Endako deposit.

The regional geology description in this section of the report has been sourced and summarised from a paper titled *The Endako Batholith: Episodic Plutonism Culminating in Formation of the Endako Porphyry Deposit, North-Central British Columbia* authored by Mike Villeneuve and Joseph B. Wallen, dated 2001. The Property geology has been reproduced with modifications from the 2006 Claim Assessment report authored by Ms. Terri Millinoff, P.Geo. and Mr. J.R. Stacey, B.Sc., of Taiga Consultants Ltd.

7.1 REGIONAL GEOLOGY

The Francois Lake Intrusions occur as numerous granitic intrusions of middle to late Jurassic age. The main body of the Francois Lake Intrusions, which is the host of the Endako molybdenite deposit, is a large northwesterly trending composite batholith that has been emplaced along the boundary zone between the Cache Creek and Stikine terrane of the Intermontage Tectonic Belt (Figure 7.1 below). The composite Endako batholith stretches from the Nechako River to south of Babine Lake and is divided into three mainly Mesozoic suites on the basis of geologic mapping conducted in the late 1990s, as follows:

- The Stern Creek plutonic suite dated at 219.3 Ma is comprised of foliated and locally gneissic, hornblende ± biotite, diorite to granodiorite that displays a distinctive through-going foliation. Grain size is variable and ranges from extremely coarse to medium.

- The Stag Lake plutonic suite forms the western, northeastern and eastern margins of the Endako batholith and is comprised of intermediate to mafic phases ranging in age from 180 Ma to 161 Ma. These include gabbro and diorite of the Boer, Stag Lake, and Twenty-Six Mile Lake phases, hornblende-biotite quartz monzonite of the Sugarloaf and Limit Lake phases and quartz monzonite to monzogranite of the Tintagel, Stellako and Caledonia phases.

- The Francois Lake plutonic suite forms the bulk of the exposed batholith and serves as host rock to the Endako deposit. The lack of penetrative deformation fabrics and the more chemically evolved felsic compositions are consistent with it being younger than the Stag Lake plutonic suite. The Francois Lake suite has been sub-divided into two sub-suites based on composition, texture, mineralogy and age. The older Glenannan sub-suite (157 Ma to 155 Ma) displays a range of composition from biotite monzogranite to hornblende-biotite granodiorite. Rocks are generally medium to coarse grained. The younger Endako sub-suite (149 Ma to 145 Ma) contains the Casey phase and the Endako phase with its Francois sub-phase. These bodies consist of medium to fine grained biotite-monzogranite to granodiorite units

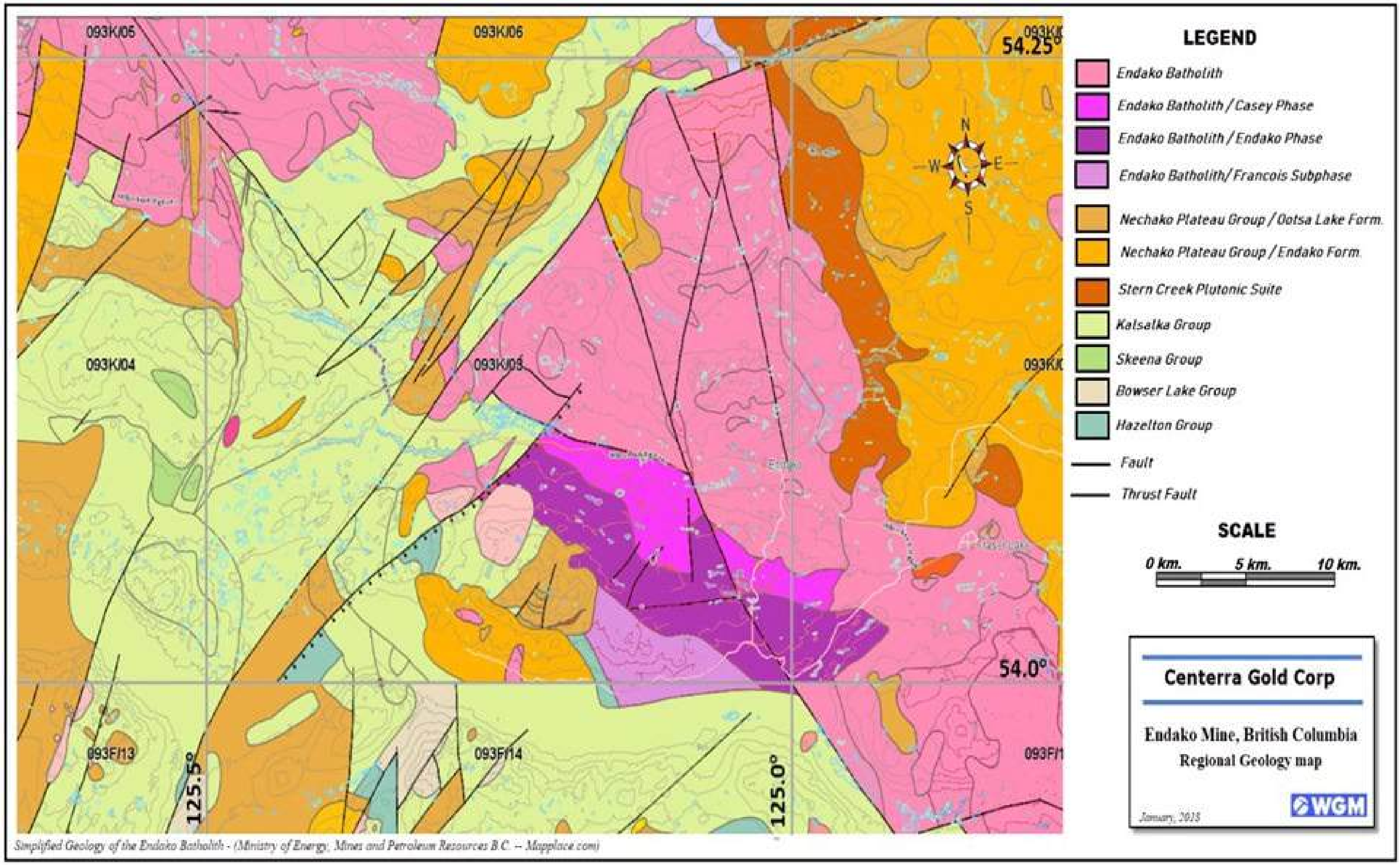


Figure 7.1 Geological Setting and Mineralisation - Regional Geological Map
Source: *WGM, January 2018*

The Endako phase of the Endako sub-suite forms a northwesterly, elongated body consisting of coarse-grained, dark pink to orange, biotite-hornblende granodiorite to monzogranite, sub-porphyritic with distinctive orange K feldspar phenocrysts. Except in the open pit, it is remarkably fresh with unaltered mafic minerals and poor veining.

The Francois sub-phase is exposed adjacent to the Endako phase and consists of purple to red, medium-grained, equigranular biotite (± hornblende) granodiorite to monzogranite.

The Casey phase is exposed only south of Highway 16 and it borders the Endako phase on its northern side. It consists of fine to medium grained, dark pink, granophyric biotite monzogranite.

7.2 PROPERTY GEOLOGY

The Endako molybdenite deposit is hosted within the Endako Quartz Monzonite, which is intruded by younger Casey Alaskite toward the north and François Granite toward the south. In the Endako Mine area, Endako Quartz Monzonite has been intruded by aplite, andesite, quartz-feldspar porphyry and porphyritic granite dykes and post-ore basaltic dykes (Figure 7.2, below).

The deposit is elongated in a northwest-southeast orientation with a maximum length of 4,800 m and a width of 750 m. The orebody is a series of major en-echelon moly-sulfide veins that strike from north through east across the deposit and dip west to south.

Structural studies that were initiated in 1973 resulted in the creation of the “Endako Vein System” concept. This model is based on the fact that a complex array of structural elements present in the Endako orebody are actually distributed along certain natural axes. When these axes are properly identified, it allows the grouping of the structural elements into natural workable units or systems. Each system, therefore, possesses a definite structural style, and because hydrothermal mineralogy is a function of structure, each system also possesses a characteristic mineralogical style.

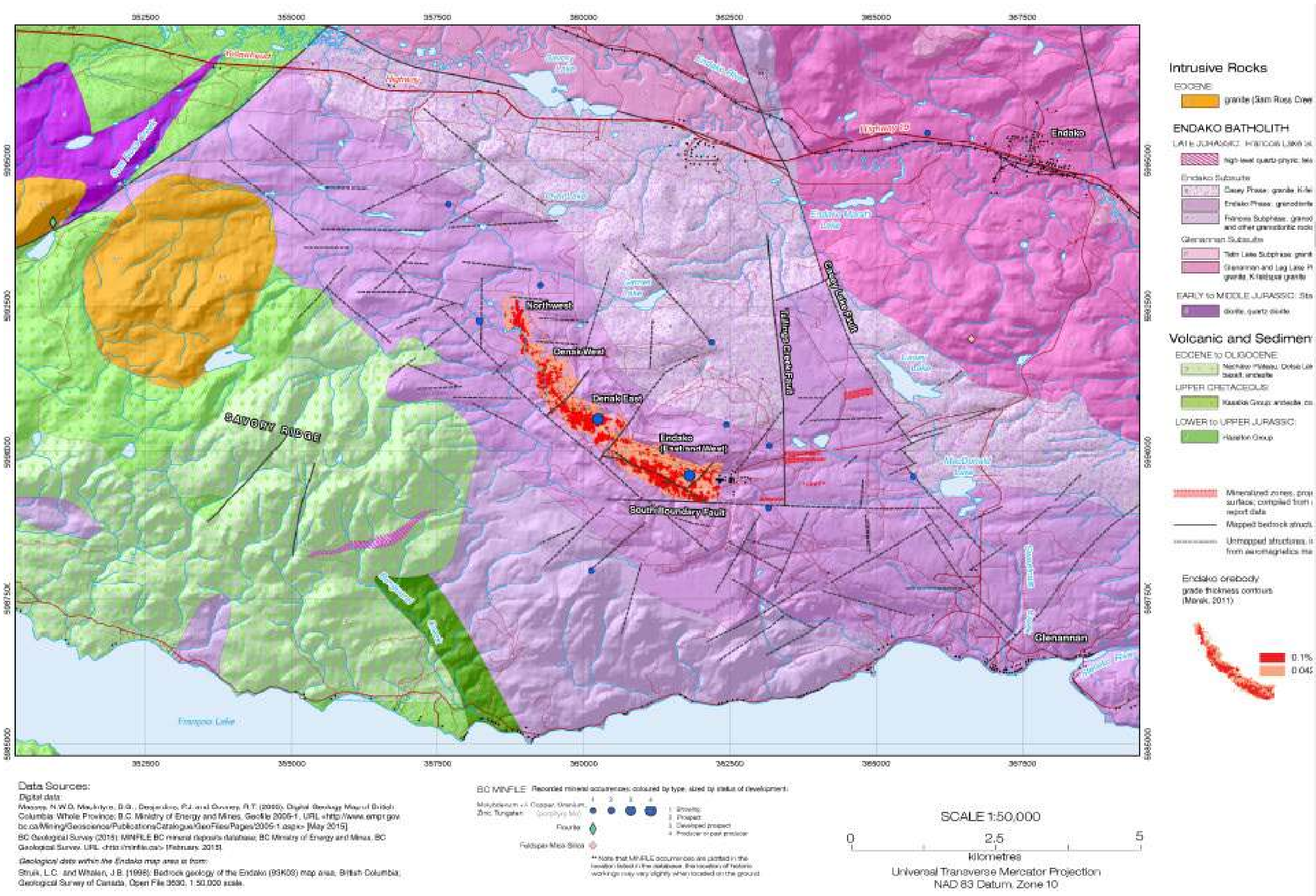


Figure 7.2 Endako Property Geology Map
***Source:** Thompson Creek Metals*

7.3 LITHOLOGY

Lithology is relatively simple in the pit area. The dominant rock type is the Endako Quartz Monzonite. This unit is intruded by barren quartz-feldspar porphyry dykes and weakly mineralised aplite dykes (see Figure 7.3, below).

- **Endako Quartz Monzonite** is the dominant rock type encountered in diamond drilling in the Endako pit. This pink to orange-pink phase is equigranular to weakly porphyritic with grain size typically 3 millimetres (mm) to 4 mm, with K-feldspar crystals ranging up to 7 mm. Its composition is typically 30% quartz, 35% K-feldspar, 30% plagioclase and 5% to 10% variably chloritized biotite. In the ore zone, the unit is variably kaolinized ranging in colour from pale greenish to creamy white.

- **Casey Alaskite** consists of 33% quartz, 40% orthoclase, 25% plagioclase, and 2% biotite. It is a fine to medium crystalline, pink or buff leucogranite, characterised by inequigranular texture, low biotite content and absence of hornblende.

- **Quartz-Feldspar Porphyry Dykes** are pale pink to tan in colour with an aphanitic or very finely crystalline groundmass with quartz and feldspar phenocrysts averaging 3 mm to 4 mm in size.

- **Aplite Dykes** are typically pink and fine to medium-grained quartz-K-feldspar-rich dykes. These dykes range up to several metres thick, show sharp contacts with host rocks and exhibit no chilled selvages. In the ore zone, aplite dykes are often mineralised with thin stockwork quartz molybdenite veinlets. Above the South Basalt Fault, aplite often hosts quartz-pyrite stringers.

- **Basalt (Andesite) Dykes** are post-ore basaltic dykes. The dark greenish grey, fine-grained and locally porphyritic dykes in the Endako pit are often associated with major fault systems. The South Basalt Fault is the best exposed fault/basalt dyke structure.

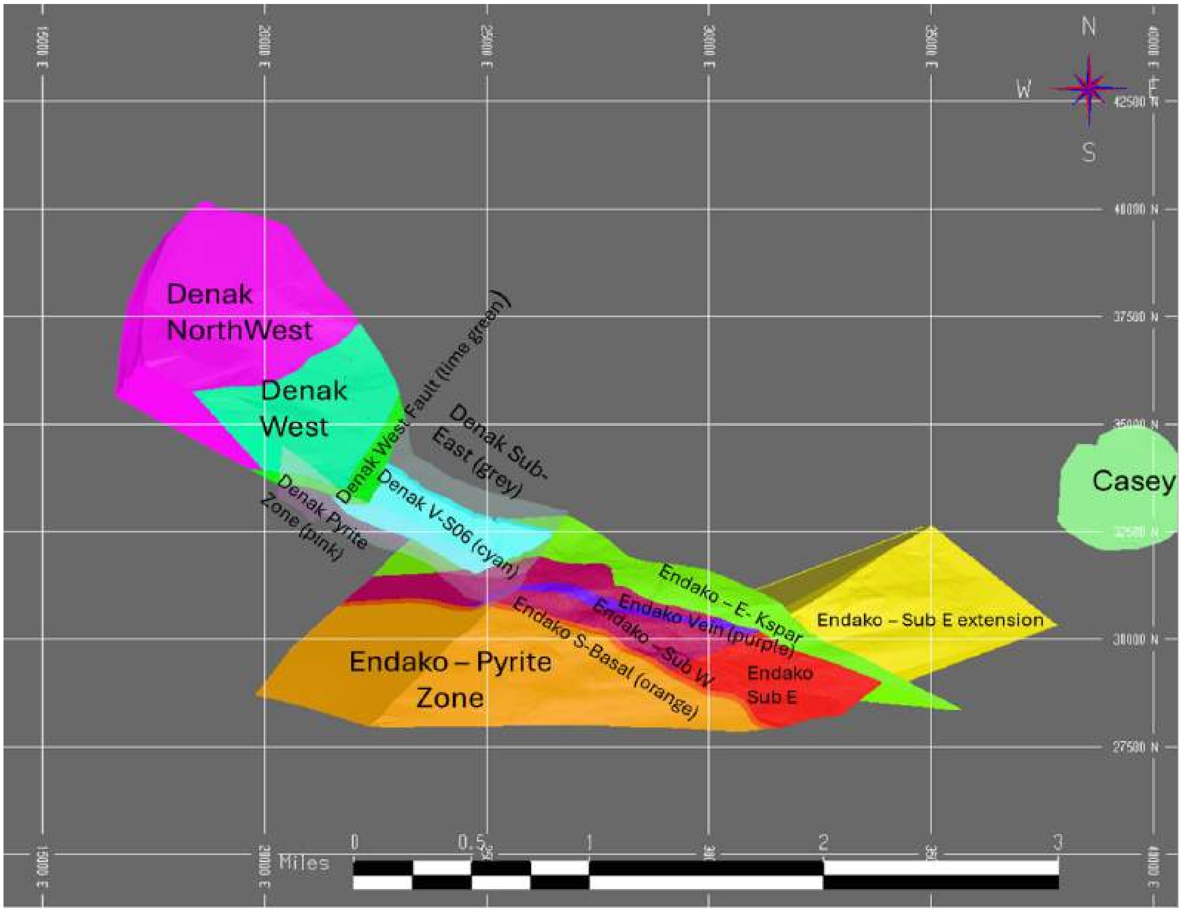


Figure 7.3 ***Detailed Lithological Units Identified by Endako Geologists***
Source: *AMPL, HXGnMinePlan™, 2025*

7.4 STRUCTURE

Pre-ore dykes that are associated with the Endako deposit strike to the northeast with vertical to steep westerly dips. These dykes have sharp contacts with little evidence of any deformation during intrusion. Post-ore basaltic dykes are marked by extensive gouge and brecciation and are associated with major structures that likely predate ore deposition. The South Boundary Fault appears to be a major controlling structure for both subsidiary structures and later hydrothermal activity.

Four structurally distinct zones have been identified from east to west: Endako East, Endako West, Denak East and Denak West/Denak Northwest. These zones are separated by steep northeast-trending structures including the eastern pre-ore dyke swarm (between Endako East and West), West Basalt Fault and Denak West Fault. A possible fault is currently inferred between the Denak West and Denak Northwest; however, this feature remains to be mapped in the field.

The Endako East Zone hosts veins that dip shallowly to the northwest. The Endako West Zone hosts veins that dip to the south; the South Basalt Fault appears to be a post-ore component of this south vein system (Bysouth and Wong, 1995). Ore structures in the Denak East pit dip southwesterly, turning abruptly to westerly dips in the Denak West pit. Secondary controls include northeast trending structures with moderate southeast dips (Figure 7.4, below).

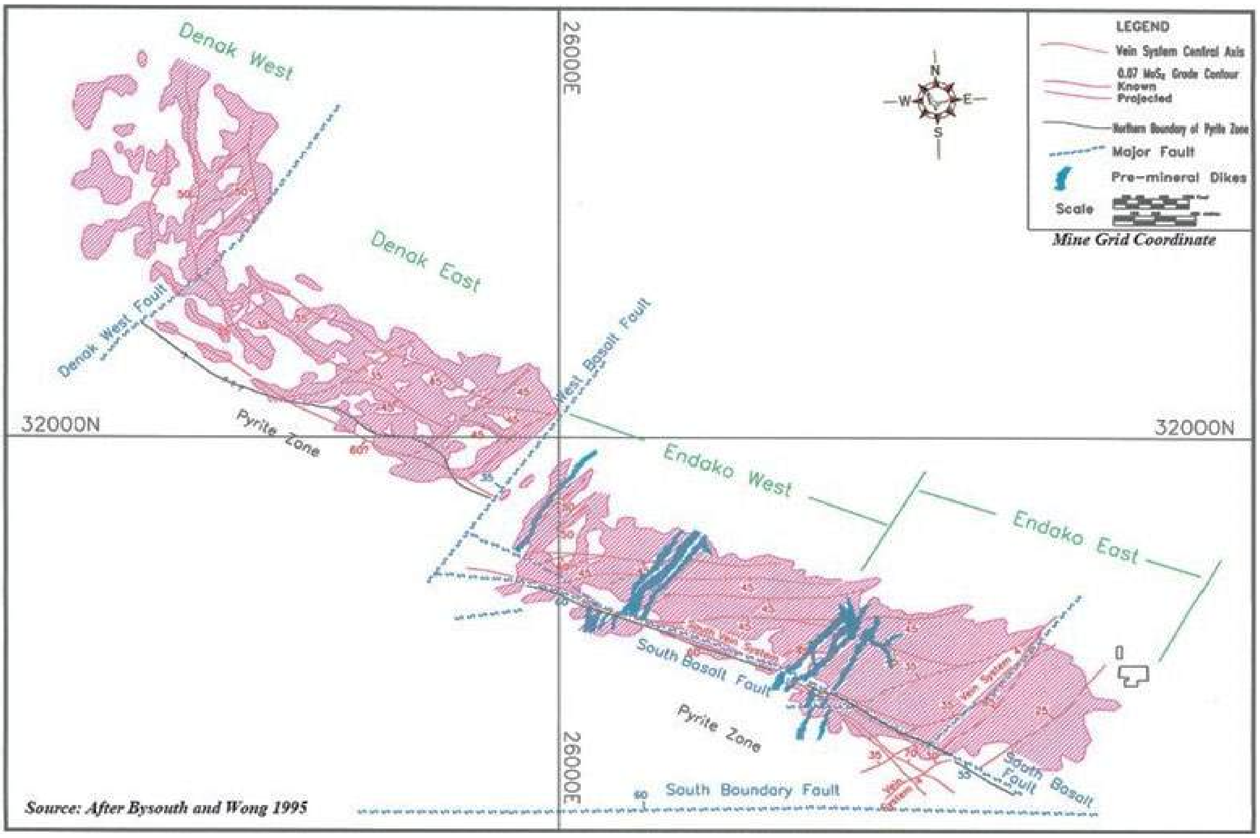


Figure 7.4 Endako Structural Domains and Vein Systems Axis

Source: *After Bysouth and Wong, 1995*

7.5 MINERALISATION AND ALTERATION

At Endako, the mineralisation consists of molybdenite with a gangue of pyrite, magnetite, minor chalcopyrite and rare bornite, bismuthinite, scheelite and specularite. The orebody consists of a series of sub-parallel or en-echelon quartz-molybdenite-pyrite veins and stockwork veins, veinlets and mineralised fractures. The increase in frequency of these veins along a preferred axis form part of the vein system concept.

Mineralisation occurs in milky white to banded or ribboned quartz veins that are often brecciated and healed by quartz and late-stage calcite and minor chalcedony. Molybdenite varies in grain size from very coarse and greasy to microscopic blue-black grains in quartz referred to as “black quartz ore”. A pyrite zone lies to the south of, and adjacent to, the orebody with a transitional boundary in the immediate hanging wall of the South Basalt Fault.

Extensive hydrothermal alteration occurs within the Endako ore zone. K-feldspar bearing envelopes develop around quartz-molybdenite veins and barren quartz veins in the footwall of the deposit. Sericite envelopes, consisting of quartz, sericite and pyrite, are developed around quartz-molybdenite and quartz-magnetite veinlets in the orebody, and quartz-pyrite veins in the pyrite zone. Argillic alteration (kaolinization) is pervasive throughout the orebody, ranging from weak to intense.

8.0 DEPOSIT TYPES

This section presents Mr. Desautels' text (reviewed and accepted by Mr. Bakker Finley of AMPL), as prepared for the unpublished Centerra Gold Feasibility Study 2022/2023.

The Endako deposit is a porphyry molybdenum deposit. The geometry and geologic occurrence are such that it is sometimes used as an example to define a style of mineralisation known as an Endako Style mineralisation. Mr. W. Sinclair, 1995, described the porphyry molybdenum deposit as a Low-F type. The following description was summarised from the mineral deposit profiles from the Ministry of Energy, Mines and Petroleum Resources.

This deposit type is characterised by a stockwork of molybdenite-bearing quartz veinlets and fractures in intermediate to felsic intrusive rocks and associated country rocks. Deposits tend to be low-grade but large and amenable to bulk mining methods. Porphyry molybdenum deposits can occur in a number of host rocks. Genetically related intrusive rocks range from granodiorite to granite and their fine-grained equivalents, with quartz monzonite most common: they are commonly porphyritic. The intrusive rocks are characterised by low F contents (generally <0.1% F) compared to intrusive rocks associated with Climax-type porphyry molybdenum deposits. Mineralisation is predominantly structurally controlled; mainly stockworks of cross-cutting fractures and quartz veinlets, also veins, vein sets and breccias.

Molybdenite is the principal ore mineral; chalcopyrite, scheelite and galena are generally subordinate. Alteration mineralogy is similar to that of porphyry copper deposits. A core zone of potassic and silicic alteration is characterised by hydrothermal K-feldspar, biotite, quartz and, in some cases, anhydrite. Phyllic alteration typically surrounds and may be superimposed to various degrees on the potassic-silicic core. Propylitic alteration, consisting mainly of chlorite and epidote, may extend for hundreds of metres beyond the zones of potassic-silicic and phyllic alteration.

The Endako deposit fits the above description. Endako is lower in silica content than the Climax-style deposits and is hosted by a monzonite or monzogranite host rather than the more rhyolitic host rocks of the Climax-style deposits. Endako is a magmatic – hydrothermal deposit associated with subduction around island arc formation or continental collision. The deposit is predominately structurally controlled, mainly as stockworks and en-echelon veinlets that establish the overall orientation of the deposit. The primary ore is molybdenite (MoS_2) with minor associated chalcopyrite, scheelite and galena. Gangue minerals are quartz, pyrite, K-feldspar, biotite, sericite, clays, calcite and anhydrite.

9.0 EXPLORATION

Exploration was ongoing from the mid-sixties through to 2016. Exploration efforts included geochemical sampling and diamond and percussion drilling. Most of the exploration effort at Endako has been by drilling. Since most drill holes completed at Endako are still relevant to the Resource estimation, the drill programs are summarised in Section 10.0 of this report. This section also notes the other exploration methods that have been completed by the Endako Joint Venture since 1997.

In 1970, Endako undertook limited Geochemical and Geophysical work in a report entitled *Geochemcial and Geophysical Report, Oval Group of Claims, Endako Mines, Ltd*. The report is available at the British Columbia ARIS government website (file 2408).

A modest geophysical survey was carried out in 1997. Details on this program were not available for review by AMPL.

In 2004, an induced polarization (IP) survey was performed at the Endako Mine. The survey was performed by Scott Geophysics Ltd. using a Scintrex IPR12 receiver and an IPC7 transmitter. A total of 12,200 ft of IP and Survey was completed along 3 lines located east of the Endako Mine and mill complex. The pole-dipole array was used for the IP survey, using a 200 ft dipole spacing and at “n” separation of 1 to 4. The online current electrode was to the south of the potential electrodes on line 1E and to the north on lines 2E and 3E. Sections were presented in the report reviewed by AMPL, but no conclusion was drawn. Details of the program are available on the British Columbia ARIS government website (file 27406).

9.1.1 Endako Mine 2013 Exploration Lidar Topographic Survey and Orthophotograph Base Mapping

In 2013, Endako conducted a LiDAR survey conducted by Eagle Mapping Ltd. of Port Coquitlam. The Endako Mine was flown from August 9 to 12, 2013 using a Cessna 206 and Riegl VQ580 LiDAR™ scanner. The orthophotograph was flown with a Piper Aztec, using a UltraCam Eagle Digital Camera.

All data was processed in the Eagle Mapping Port Coquitlam office using TerraSolid LiDAR™ processing software. The imagery was also processed in the Port Coquitlam office using Trimble OrthoMaster/OrthoVista™ software.

The LiDAR point cloud was surveyed and processed to produce a point density of at least, but usually exceeding, 2 to 4 points per square metre. The accuracy of the LiDAR points was ± 15 centimetres (cm) vertically and ± 30 cm horizontally on hard open ground. The topography map was compiled to 1 m accuracy.

In 2013, Endako also embarked on a project to complete a mineral title legal survey in support of a mineral claim to mineral lease application. Detailed research and confirmation of the mineral titles held by Thompson Creek (Endako Mines) was to be completed and followed by a legal ground survey to define, blaze and monument the proposed new lease area. The contract was awarded to Focus Corporation of Prince George, British Columbia. Filed work took place from March to November 2013.

In 2013, Endako Mine requested the Integral Ecology Group (IEG) design and implement a revised reclamation assessment program. The request followed the recognition that the historic assessment program was no longer well aligned with current and regulatory objectives, particularly with an increasing emphasis on "ecosystem-based" criteria for reclamation assessment, and thus, did not provide the information necessary to evaluate reclamation success with respect to these objectives.

The 2013 reclamation assessment program was designed as a "pilot" study to test new methods of data collection and interpretation and to review the results with regulators prior to a full-scale program implementation. Key elements of this redesigned program included using permanent sample plots on both reclaimed and adjacent "reference" areas to provide information on local non-mined conditions and on the convergence of reclaimed ecosystems with these conditions.

In 2015, 27 reclaimed plots and 3 reference plots were installed on 8 reclamation sites and 3 references sites. This brought the total number of plots installed from 2013 to 2015 up to 78 (62 reclamation and 16 reference plots). In addition, 6 reclaimed plots on 2 reclamation sites were re-sampled in accordance with the re-sampling schedule proposed in the IEG report. The 2015 IEG report presents a combined analysis of results from the three study years and evaluates reclamation success at the reclaimed-site level and at the Mine level based on the Criteria and Indicators (C&I) framework proposed in the 2014 IEG report.

In 2016, Endako conducted a small field program from September 14 to September 27. A total of 59 soil, 6 rock (grab) and 1 stream sediment samples was collected in an area north of the Denak Northwest extension (Figure 9.1, below). Assessment Report Endako Mine 2016 Geochemical Survey Mineral Claims 1039549, 507254, 1017773

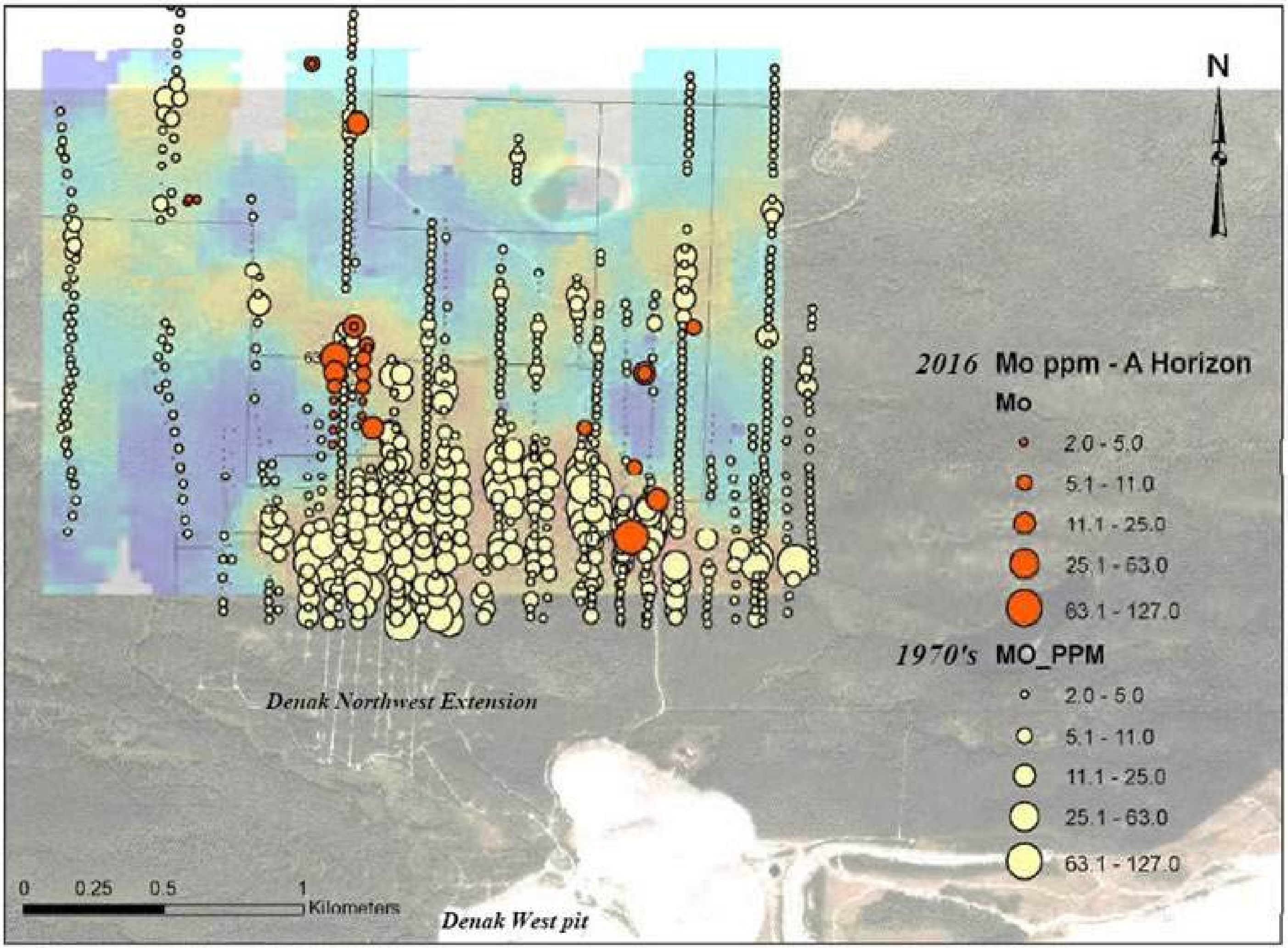
At each site location, an attempt was made to collect samples from three soil horizons – A, B and Ah. Soil sample test pits were dug to depths from 15 cm to 30 cm below the humus. The main soil sampling occurred two lines, 100 m apart and with 50 m sample spacing.

All rock samples collected were grabs of typical Endako Quartz Monzonite located adjacent to a soil test pit. No visible sulphides were noted. One stream silt sample was taken from Watkins Creek.

The soil sample data was combined with the soil samples collected in 1970. Results from the program confirmed molybdenum anomalies identified from the property scale sampling programs in the 1970s. There does not appear to be any location bias to the historic results.

The program recommended that additional soil surveys be conducted. It was also recommended that, prior to initiating a larger scale sampling program, a soil geochemical orientation survey be completed over the known resource northwest of the Denak West pit.

In 2020, a report titled, *Assessment Report #38515, A Report on Geophysical Surveying – Endako Mine*, Fraser Lake Area, British Columbia, 2020, Alex Walcott, B.Sc. was filed for assessment with ARIS.



Source: Assessment report 2016 Geochemical Survey (2016)

Figure 9.1 2017 Soil Sampling Program

Source: *Assessment Report 2016 Geochemical Survey, 2016*

10.0 DRILLING

This section presents in portion Mr. Desautels' text (reviewed and accepted by Mr. Finley Bakker of AMPL), as prepared for the unpublished Centerra Gold Feasibility Study 2022/2023. It is further augmented by data gathered by Mr. Bakker on his site visit in September 2025.

The Endako drill hole database consists of historic exploration and definition drilling, mixed with more recent drilling. This database was supplemented by production blast holes, which were also used in the grade estimation. Only one drill report, prior to 1989, was available for review by AMPL. This early 1969 report covered 4 holes drilled on the Casey Zone, and the remaining holes were drilled west of the Casey Zone in an area now covered by the tailings pond. This section will focus on the information gathered by AMPL, from claim assessment reports filed with the B.C. Government since 1980, and, for the most recent drilling, Endako Mine drill reports.

10.1 PLACER DOME PERCUSSION DRILL PROGRAM 1978

A series of vertical two-inch diameter percussion drill holes were drilled on the Casey Creek/Denak Northwest areas. Percussion drilling was contracted to Josco Mining Company Ltd. and site preparation was contracted to Pooley Construction Company Ltd. Holes were sited in the field using tape and compass from known survey stations. Sludge samples were recovered from the holes and assayed at the Endako Laboratory. The holes were imported into the HxGN MinePlan™ 3D database and are identifiable with the "R" prefix. These holes were used for the resource estimate as they were considered by the QP as still being reliable in terms of grade and although most have since been superseded with diamond drilling.

This accounts for 330 holes or approximately 20% of drill hole data available for resource modeling (excluding blasthole data).

10.2 PLACER DOME DRILLING AND CORE HANDLING PROCEDURES

Little is known about the drill programs prior to 1989. The earliest hole in the database is Hole S-39, which was drilled in the early 1960s. Hole S-48 was drilled in 1962. For this hole, the drill log indicated the core size was AXF at the beginning of the hole and changed to AX (1-3/16 inches) after going through the overburden. Hole S-52 was drilled in 1962 and the log indicated a BX (1-5/8 inches) core size. The author of this report was employed by Placer Dome between 1980 through to 1982. During that period, the core size was NQ (1-7/8 inches) and the drilling and core handling procedure was similar to the procedures described below.

During the period covering 1989 through to 1997, the drilling was carried out by L.D.S. Diamond Drilling Ltd. of Kamloops, British Columbia. The holes were bored by diamond drilling recovering NQ wireline core, which has a nominal size of 1-7/8 inch in diameter.

A Sperry Sun instrument was used to measure drill hole deviations during the 1989 drill campaign but the assessment reports for subsequent years do not itemise any expenses for a downhole survey tool. Since there is no downhole survey information in the mine database that was provided, it is assumed that no downhole deviation measurements were collected after 1989.

Only about 25 holes had any downholes surveys undertaken, less than 2% – normally this could be considered an issue but the mine always followed up diamond drill holes with blast hole sampling and this is used in final determination of blast outlines – it is anticipated that this procedure will continue and as such, the QP is satisfied with historical procedure but would recommend downhole surveys for future work in particular for exploration drilling (see Figure 10.1 to Figure 10.5, below).

Figure 10.1 Placer Dome Drill Program – Original Paper Copies "S" Series Holes

Source: *AMPL, 2025*

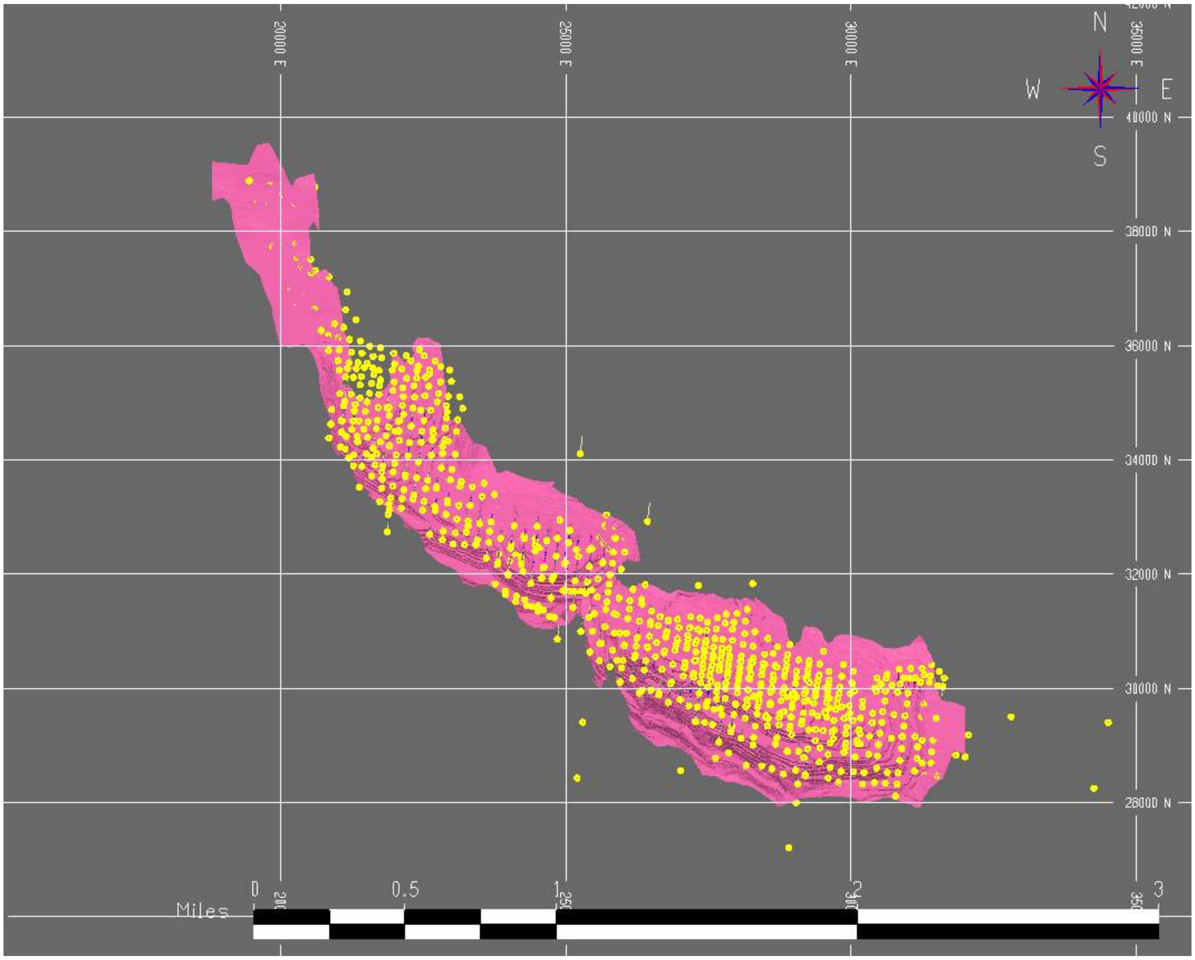


Figure 10.2 Placer Dome Drill Program – "S" Series Holes
Source: *AMPL, 2025*

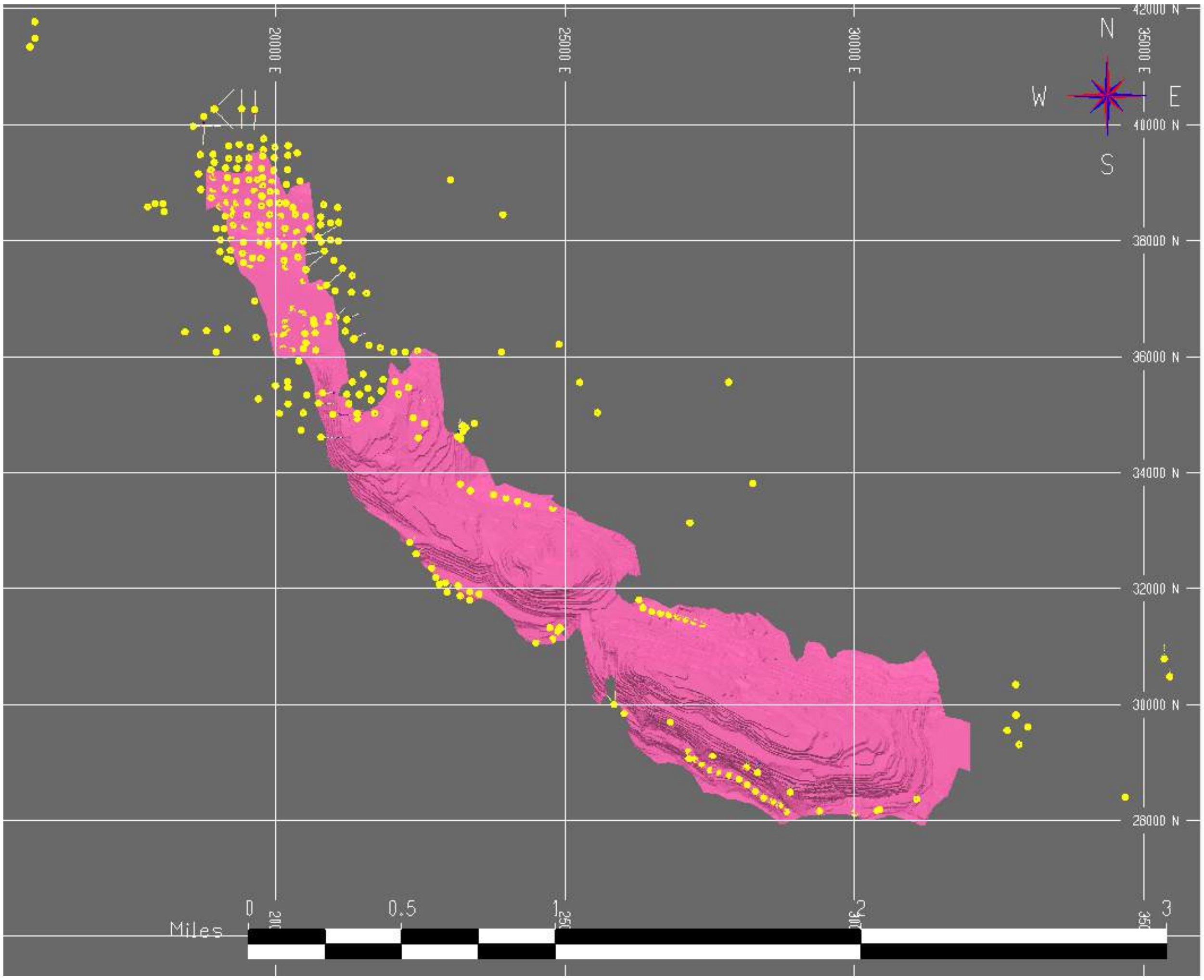


Figure 10.3 Thompson Creek Drill Program
***Source:** AMPL, 2025*

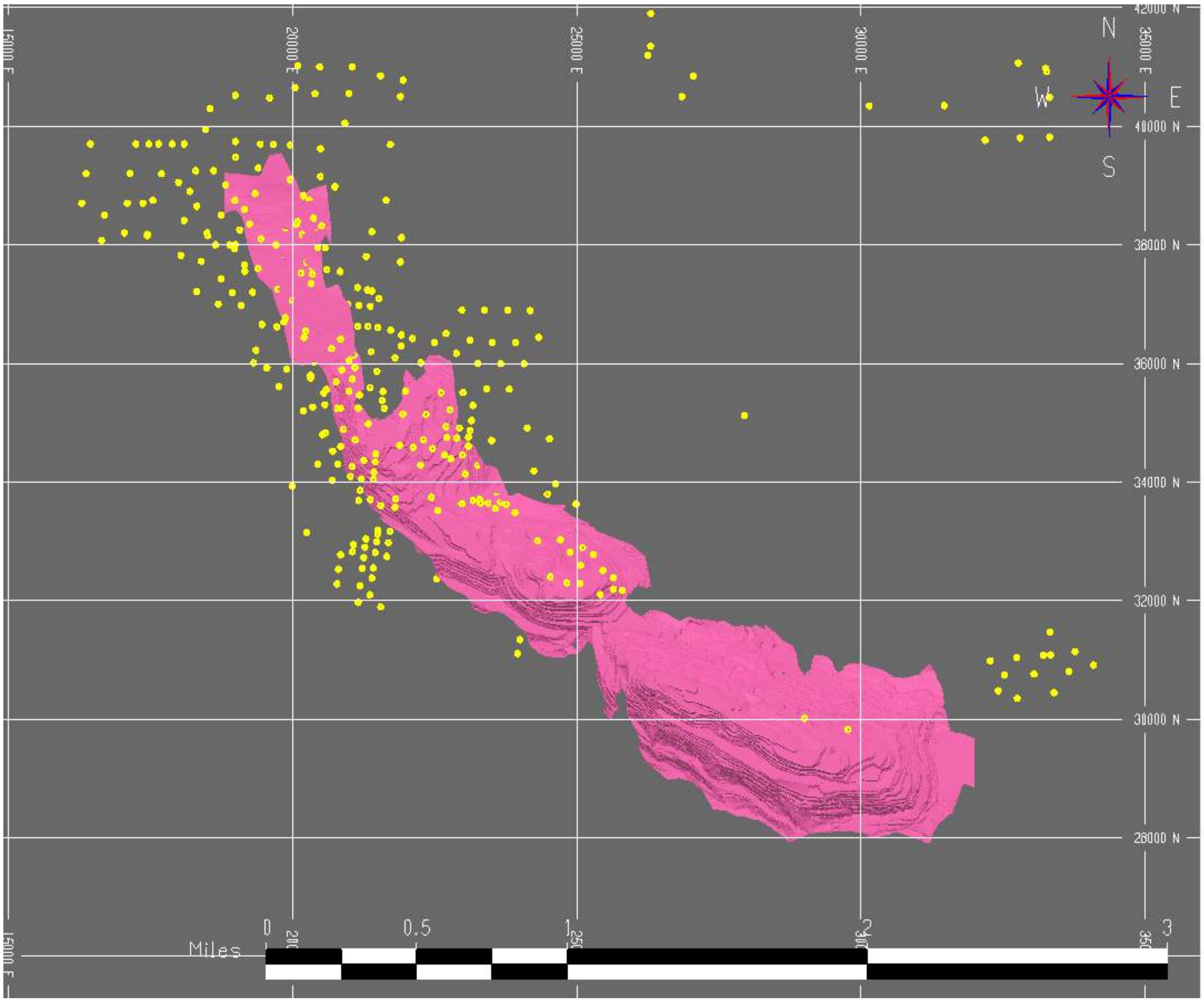


Figure 10.4 RC Drilling Program
***Source:** AMPL, 2025*

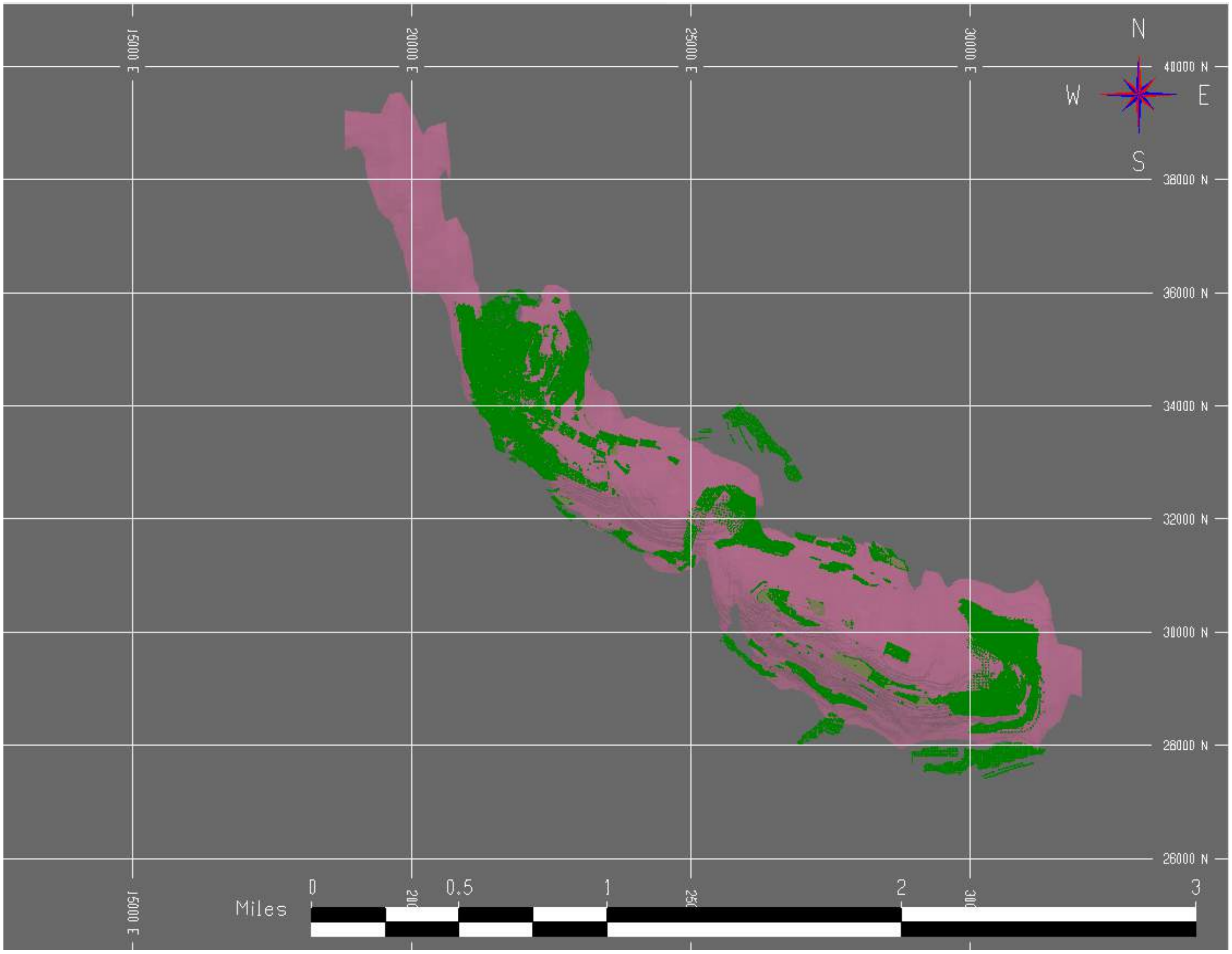


Figure 10.5 Later Portion of Blastholes in Green Superimposed on Resource (31,784 holes)
Source: *AMPL, 2025*

The samples were all handled by Endako personnel. Core was boxed at the rig by the drill contractor, but after transport of the core to the Endako logging and assay facilities, the diamond drill samples were within 100% control of the Endako staff.

Drill core was geologically logged on a 1 inch = 10 ft graphic log and was sampled in corresponding 10-ft intervals for assaying. The information recorded consisted of rock types, alteration, mineralisation and structures. The rock quality information was also recorded for each 10-ft interval. This information consisted of a fracture frequency distribution, rock quality designation (RQD), core recovery and specific gravity on the earlier holes (prior to 1979).

The 10-ft interval core samples were not split. The core was bagged whole and transported from the core logging facility located at the Endako Mine site to the Endako Assay Laboratory located in the adjacent building to the core logging facility. All sample preparation and assays were conducted at the Endako Mine Assay Laboratory, as described in Section 11.0 of this report.

All core logging and sample collection was performed by the Endako Mine staff.

10.3 PLACER DOME DIAMOND DRILLING (PRIOR TO 1989)

Little is known about the drill programs prior to 1989. Drilling was focused on expanding the resources at the Endako, the Denak East and the Denak West pits. Exploration during that period was mostly confined to the Denak Northwest extension, which was discovered during the 1978 percussion drilling program.

10.4 PLACER DOME DIAMOND DRILLING (1989 – 1997)

There are good records for the various drilling programs covering the Placer Dome drilling from 1989 to 1997. As mentioned previously, the information listed in this sub-section was extracted mainly from the B.C. government assessment reports.

10.4.1 1989 Program

Fourteen NQ (1-7/8 inch) wireline diamond drill holes (S-656 to S-669) totalling 11,407 ft were drilled in the Denak East Open Pit, along the south wall of Endako and Endako West Open Pits and within the Endako West Pit. The program was designed to test for the possible extension to the Denak East and Endako Ore Zones and for geotechnical information.

10.4.2 1990 Program

Thirty-one NQ wireline holes (S-670 to S-701) totalling 13,200 ft were drilled within the Southeast portion of the Denak East Zone.

There were no assessment reports covering the holes from S-670 to S-701.

10.4.3 1991 Program

Twenty-two NQ wireline holes (S-701 to S-722) totalling 9,388 ft were drilled within the Northeast portion of the Endako Open Pit and along its highwall crest primarily for delineation of possible extension to the Endako Ore Zone.

10.4.4 1992-1993 Program

Forty-four NQ wireline holes (S-723 to S-766) totalling 20,282 ft were drilled within the Denak West Open Pit, along its periphery, and toward the Watkins Creek area for ore control refinement and for delineation of possible extensions to the Denak West Ore Zone.

For the program from 1989 through to 1993, Endako reported that quartz, molybdenite and associated ore minerals occur in random oriented fractures in a stockwork adjacent to, and surrounding, quartz molybdenite veins, which are 15 cm to ½ m wide. The occurrence of such molybdenite veins was rare and the associated stockwork was weak to non-existent.

10.4.5 1994-1995 Program

Eleven NQ wireline holes (S-775 to S-785) totalling 4,781 ft were drilled in the Watkin's Creek area northwest of the Denak West Pit. These holes were drilled to improve the delineation of the molybdenum mineralisation along strike from the Denak orebody. Eight more holes (S-786 to S-793) totalling 2,996 ft were drilled on the south wall of the pit on Bench 2750, and on the south ramp of the Endako pit for a total of 7,777 ft.

Endako reported intense shearing in the holes covering the Watkin's Creek area. Drilling indicated a north-south trending post mineral fault that may offset the Denak West veins in a right-lateral fashion. They also reported that mineralisation occurs primarily in sheared quartz veins and gouge with pyrite, chalcopyrite, magnetite and hypogene hematite as common accessories but without significant potassic alteration. Significant mineralisation reportedly occurred in a 200 ft to 300 ft wide, northwest trending zone.

The Endako south wall drilling showed that mineralisation was more restricted, and of higher grade, than what was modeled at the time.

10.4.6 Post 1995

There were no assessment reports covering the holes from S-794 to S-856. Some of these holes were likely drilled in 1997 under Thompson Creek management. For the purpose of this report, all the "S" series holes are considered to have been drilled by Placer Dome. The drill program(s) mostly focused on extending the resources east of the Endako Pit toward the engineering office. Many holes were also drilled to improve the resources in the Denak West Pit and refine the extent of the mineralisation on the east wall of the pit. Exploration drilling was carried out east of the Endako Pit toward the Casey Zone.

10.5 THOMPSON CREEK DRILLING AND CORE LOGGING PROCEDURES

Drilling was carried out by Hy-Tech Drilling Ltd. of Smithers, British Columbia during the 2003-2004 drill campaign. L.D.S. Diamond Drilling Ltd. of Kamloops, British Columbia was used from 2001, 2002 then in 2006 through to 2011. The holes were bored by diamond drilling recovering NQ wireline core, which has a nominal size of 1-7/8 inches in diameter.

Drill collar locations were surveyed prior to drilling with the Trimble GPS™ system used for surveying on the Endako Mine site. A stake was placed at the planned collar location, and, if the drill hole was to be inclined, several stakes were added for use as front sites and back sites. Once the diamond drillers were set up and ready to drill, the azimuth and inclination of the mast was checked with a Brunton™ compass by Endako personnel.

There was no mention of down-the-hole surveys for the 2001, 2002, 2003, 2004, 2006 and 2007 drill programs in the report reviewed; although, occasional acid tests were done on inclined holes during the 2006 drill program. In 2008, a FLEXIT™ Smart Tool was used to record the down-the-hole deviation on inclined drill holes, which mark the first time where a downhole survey tool was routinely used. This instrument was replaced by the PeeWee™ survey instrument obtained from Equipment JexPlore Inc. for the 2010 and subsequent drill programs.

The samples were all handled by the Endako personnel. Core is boxed at the rig by the drill contractor, but after transport of the core to the Endako logging and assay facilities, the diamond drill samples are within 100% control of the Endako staff or its independent geological contractor.

Logging procedures remained more or less the same since the Placer Dome era. The drill core was geologically logged on 1 inch = 10 ft graphic logs and was normally sampled in 10-ft intervals but could vary between 2 ft and 18 ft. The information recorded consisted of rock types, alteration, mineralisation and structures. The rock quality information was also recorded for each 10-ft interval. This information consisted of a fracture frequency distribution, rock quality designation (RQD) and core recovery.

The core samples were typically split using a manual splitter with half the core put in plastic bags for delivery to the assay lab and the other half retained for future reference. The remaining half of the core was stored in the core storage area at the Endako Mine site, while pulps were stored in the core logging facility. Core prior to 2006 has been lost and is no longer available for review. The core intended to be sent for metallurgical testing was fully sampled and not split. All sample preparation and assays were conducted at the Endako Mine Assay Laboratory as described in Section 11.0 of this report.

A Quality Assurance and Quality Control (QA/QC) program was implemented during the 2008 drill campaign. This program was slightly modified prior to the 2010 drilling campaign and the revised program remained in place for the 2011 drilling. Details of these programs are discussed in Section 11.0 of this report.

Core logging and sample collection was mostly performed by various independent contractors to Thompson Creek, with a minority of the holes logged by the Endako geological staff.

10.6 THOMPSON CREEK DRILLING (2001 – 2011)

10.6.1 2001 Program

Five diamond drill holes were completed totalling 2,535 ft on two targets. Three holes were completed in the water tank area to the northeast and two more holes were completed in the southeast dump area. Rare MoS_2 was encountered in fresh quartz monzonite in the southeast dump area. In the water tank area, increased structural complexity and significant though sub-economic molybdenite mineralisation, was encountered. An intercept of 0.132% MoS_2 over 10 ft. (3.05 m) in the water tank area was recommended for follow up. The core logging and sample collection was performed by Mr. Christopher Wild, P.Eng.

10.6.2 2002 Program

Fourteen diamond drill holes totalling 5,166 ft were completed along the south wall and at the bottom of the Endako Pit on two targets. Holes 02-01, -02, -03, -04 and -05 were in-fill drill holes intended to improve the resource model or add confidence to the grade distribution. Drill holes 02-06 to 02-14 were drilled to improve geological and metallurgical data to determine the feasibility of a proposed south wall pushback. The core logging and sample collection was performed by Mr. Christopher Wild, P.Eng.

10.6.3 2003-2004 Program

Seven diamond drill holes totalling 2,580 ft were completed in January 2004. Three holes tested a subtle IP chargeability high 3,000 ft east of the Endako Pit. Anomalous but sub-economic, MoS_2 was found in two of the holes. Three additional holes tested the economic viability of the northeast side of the Denak Pit. One hole was abandoned in glacial till. Grades in these holes were reportedly near economic value, core logging and sample collection was performed by Mr. Christopher Wild, P.Eng.

Phase 2 of this drill program continued in 2004 with four additional holes totalling 1,948 ft. Holes were targeted at an area north and east of Hole 04-05, located approximately 3,000 ft east of the Endako Pit. The core logging and sample collection was performed by Mr. Daryl J. Hanson, P.Eng.

There was no drilling in 2005.

10.6.4 2006 Program

Taiga Consultants Ltd. (Taiga) was contracted to log drill holes 06-06 to 06-35 and report on the results from the 35-hole drill program. Thirty-two of these; holes 06-01 to 06-05, 06-08 and 06-10 to 06-35 were reported in the 2006 Assessment Report. Core was logged by Ms. Terri Millinoff, B.Sc., P.Geol., Mr. J.R.Stacey, B.Sc., of Taiga and Mr. D.J. Hanson, P.Eng. The total length of NQ sized core generated from the drilling program reported in the Assessment Report was 16,870 ft. This exploration drill program was designed to delineate additional molybdenite resources in the vicinity of the mining operation. Holes targeted the Denak West Pit, the Denak East Pit and the Casey Lake area.

The program was expanded to add an additional 32 holes totalling 16,509 ft. The program targeted the Casey Lake area, Denak West and the edge of the Denak East Pit area.

10.6.5 2007 Program

The 2007 diamond drill program was designed by the Endako personnel and Mr. P. Mudry of Taiga. The intent of the program included testing zones encountered in previous drill programs (particularly 2006), drilling new targets and completing condemnation drilling of Endako's proposed mill expansion site and third tailings pond area.

The program consisted of 66 holes totalling 35,846 ft. Of those, 33 holes targeted the Casey Zone, 17 holes tested targets in the Denak Area, 6 holes in Endako Pit area and 10 holes for condemnation drilling.

10.7 CASEY ZONE

Mineralisation in the area between Tailings Dam #1 and Casey Lake was first intersected by diamond drilling in 1968-1969. Ore grade molybdenite mineralisation was encountered in several holes in this area, but only as local narrow intersections. At the time, when compared to the high-grades and large tonnage known in the Endako Pit, the intersections encountered near Casey Lake were deemed too small to merit further exploration. Interest in this area re-surfaced in 2006, after Mr. Roger Steininger, a consulting geologist, was commissioned to outline potential targets for exploration near the existing mine. Results from the late 2006 drill program were encouraging and in 2007, an additional 33 holes were completed. The mineralisation appears to be strongest along an east-northeast trend, possibly representing the intersection of two or more major vein systems. Moving away from the central axis of the deposit, mineralisation remains locally strong but becomes more widespread. Depth of waste rock and overburden typically increases to the north and east.

10.8 DENAK AREA

Prior to 2006, intermittent diamond drilling programs had intersected ore grade mineralisation in several holes over the Denak northwest area, but spacing was wide and no continuity had been established between those intersections and mineralisation in the Denak West Pit. Following up on encouraging results from the 2006 program developed by Mr. R. Steininger, a total of 17 holes were added to the existing drilling. Holes 07-50 to 07-54 encountered high-grade molybdenite veins. Results from these holes, and the data from historical drilling, seem to exhibit a significant west or northwest dipping system of high-grade veins. Holes 07-55 to 07-58 were poorly mineralised, and these holes provided little reason to continue drilling to the east of the known mineralisation in the Denak area. Holes 07-59 to 07-66 were drilled to test for the continuation of molybdenite veins west of known occurrences. The holes encountered sporadic high-grade intersections.

10.9 EXPLORATION DRILLING

Six holes (07-34 to 07-39) were drilled between the Endako Mine and MacDonald Lake (Endako East). These holes were to test coincidental geochemical and geophysical anomalies. Minimal molybdenite mineralisation was intersected.

10.10 CONDEMNATION DRILLING

On the site of Endako's proposed upgraded mill and stockpile, five diamond drill holes (07-40 to 07-44) were drilled for condemnation purposes. Low-grade molybdenite mineralisation was encountered in holes 07-40, 07-42 and 07-44 but was not significant enough to warrant re-assessment of the proposed expansion.

Within the planned extent of Tailings #3, five condemnation holes (07-45 to 07-49) were drilled. Mineralisation was restricted to local trace quantities of molybdenite in quartz veins.

10.11 2008 PROGRAM

Endako's 2008 diamond drilling program involved two phases; the first phase was exploration drilling on satellite targets located within 5 km of the mill site and the second phase was in-fill drilling along the margins of the existing pits.

The initial program was divided into 2 phases amounting to 16 holes totaling 10,279 ft of diamond drilling on targets near the Endako Mine site as follows.

10.11.1 Phase IA – Denak Northwest Extension

A total of 11 vertical holes (6,996 ft) were drilled beyond a group of drill holes with promising intersections located to the northwest of the Denak West orebody. The 2007 program only tested for continuation of this mineralisation to the east and found mineralisation of little interest. Historic percussion drilling and soil sampling in the western portion of that area indicated a wide area of potential interest.

Of the 11 holes drilled, 9 holes completed along the west and north margins of the target returned significant ore grade and low-grade intersections, and several also contain promising high-grade veins. Two holes (08-40 and 08-43) were drilled near historic drill holes to fill in significant gaps in coverage.

Results from the drilling program indicated that the Denak Northwest Extension currently appears to be less laterally extensive than the Denak West deposit, and ore grade mineralisation continues to greater depths. All holes drilled to the end of the 2008 campaign complied with this basic model; however, several holes along the western margin (most notably 08-33, 08-35 and 08-36) exhibited significantly lower grades than their neighbors. Proximity and similarity of mineralisation implied the Denak Northwest Extension is part of, or closely related to, the Denak West deposit. Watkin's Creek, a northwesterly trending drainage separating the Denak West and Denak Northwest Extension, may be a surface expression of a sinistral strike-slip fault with 2,000+ ft of offset, based on the relative position of the deposits.

10.11.2 Phase IB – Casey Zone

A total of 5 vertical holes (2,846 ft) were drilled at the southwest extents of the Casey Zone, to step out from the known deposit. This brings the total drilling in the area since 2006 to 36,492 ft in 65 holes. All holes drilled in 2008 encountered significant ore grade mineralisation. Based on the data collected from 2006 through to 2008, the core of the Casey deposit appears to be a northeast-southwest trending zone where several vein systems converge; holes drilled along that zone produce lengthy and continuous ore grade intersections. These vein systems diverge to the northwest and southeast; where high-grade veins become increasingly sparse, with larger zones of waste between them.

10.11.3 Phase II – In-Fill Drilling

The subsequent phase of drilling placed 36 in-fill holes totalling 17,139 ft along the margins of the Endako, Denak East and Denak West Pits. Targets for drilling were determined by studying Endako's database of diamond drill holes and the block model developed by Wardrop Engineering Inc. in 2007. This study identified several conspicuous gaps in drill hole coverage, primarily at the margins of the existing pits, as candidates for new drilling. New data generated from this drilling was to be used to improve the Endako resource model in areas that previously contained little or no information.

All core for this drilling program was logged and sampled by the Endako geological staff.

There was no drill program in 2009.

10.12 2010 PROGRAM

The 2010 program comprised 45,202 ft of NQ diamond drilling in 91 holes. A total of 82 holes (39,816 ft) were drilled in the Denak Northwest Extension to improve the classification from Probable Reserve to Proven Reserve. Additionally, 5,386 ft were drilled in the south wall of the Denak East Pit to better refine the block model and improve grade reconciliation. Five holes in the Denak Northwest Extension were designated for geotechnical studies under consultation with Golder Associates Ltd. The 2010 Assessment Report, authored by Mr. M. Pond, P.Geo., indicated anomalous molybdenite mineralisation, with minor K-feldspar alteration and moderate to intense argillic alteration, was encountered in veins/veinlets and on fracture surfaces in all holes of the 2010 drill program. The Denak Extension is interpreted to be a continuation of the major porphyry system of the Denak West Pit. Significant mineable widths were intersected in 61 of the 82 holes drilled in the Denak Northwest Extension with the highest-grade intersections of the program in holes 10-035, 10-034 and 10-064, with 44 ft average bench grades of 0.398%, 0.363% and 0.363% MoS_2, respectively. Taiga was contracted to provide geologists to conduct the geological core logging for the drill program. Taiga logged 76 of the holes and Mr. Pond logged the last 6 holes of the program.

10.13 2011 PROGRAM

The 2011 diamond drill program was conducted from April 26, 2011 to April 10, 2012 and included several phases. Table 10.1, below, indicates the location of the drill targets.

TABLE 10.1
DIAMOND DRILL PROGRAM – 2011

Phases	Description/Location	Hole Numbers
Phase IA	Denak Extension	11A-001 to 11A-048
Phase IB	Endako Pit South Wall	11B-001 to 11B-013
Phase IIA	Denak Extension	11A-049 to 11A-052
Phase IIB	Endako Pit North Wall	11B-014 to 11B-022
Phase III	Georgia West Option	11A-053 to 11A-067

Source: *WGM, 2018*

Phase 1, comprised 28,461 ft of NQ diamond drilling in 45 holes, was completed between April 26 and December 6, 2011. This phase of the program was designed to in-fill and increase the reserve tonnes of the Denak Northwest Extension area. There were 22 significant intersections from 15 drill holes, comprised of bench composites grading greater than 0.10% MoS_2, including 5 intersections grading greater than 0.2% MoS_2.

Phase 2, comprised 1,670 ft of NQ diamond drilling in 2 holes, was completed between August 4 and August 30, 2011.

Mineralisation is still open to the north and west of the Denak Northwest Extension and may be open to the east. Continued exploration was recommended to find the limit of the mineralisation.

10.14 2012 PROGRAM

The 2012 drill program consisted of 17 in-fill holes totalling 5,009 ft to test the mineralisation below the floor of the Denak West Pit. The program outlined mineralisation exceeding 250 ft in the northern portion of Denak West. To the east, the mineralisation was weak and extended no more than 100 ft below the pit floor. Assays above 0.1% MoS_2 were rare.

No exploration program was conducted for 2013 and 2014, when the mine was shut down (see Figure 10.6 to Figure 10.9, below).

Figure 10.6 Barrels and Sea Cans Containing Rejects from Thompson Creek Diamond Drill Program

Source: *AMPL, 2025*

Figure 10.7 Diamond Drill Core from Most Recent Drill Programs

Source: *AMPL, 2025*

Figure 10.8 Boxes in Administration Building Containing Pulps from Thompson Creek Diamond Drill Programs

***Source:** AMPL, 2025*

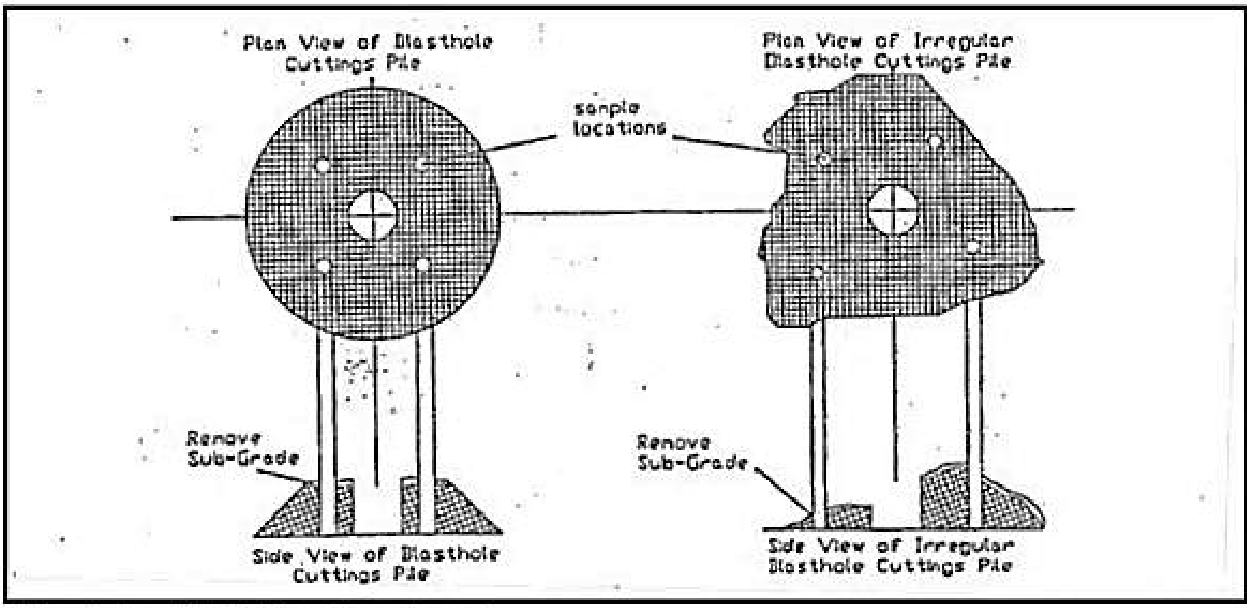
Figure 10.9 Diamond Drill Core from earlier drill programs

***Source:** AMPL, 2025*

10.15 BLAST HOLE SAMPLING PROCEDURES

It was reported that blast hole sampling procedures at Endako did not vary much throughout the years. WGM reported the procedures are described as follows:

- Two inches of drill cuttings are removed from the top of the piles to account for the sub-grade unless the survey department flagged the hole as having no sub-grade; and
- A total of four pipe cuts from each pile are collected, one for each quadrant of the pile (Figure 10-3, below), and the pipe should be inserted down to the bottom of the pile.



Drawing from Endako Mine (date unknown)

Figure 10.10 Blast Hole Sampling Procedure
***Source:** Drawing from Endako Mine (date unknown)*

The blast hole sampling program is occasionally audited by the Endako Mine staff. During the audit, the sampling procedure is reviewed with the drilling crews and 10% of the samples are sent for re-analysis. WGM reviewed the results from the 2012 audit. The chart produced for the audits indicated good reproducibility between the first sample when compared to the second sample. The linear regression (R-squared) ranges between 0.91 and 0.96 with a slope ranging from 0.81 to 0.96. There was a fair amount of scattering about the parity lines; however, no systematic bias was observed.

10.16 QUALIFIED PERSON’S COMMENTS

AMPL is of the opinion that the drill hole orientation was found to be appropriate for the deposit style and the orientation of the mineralisation.

Drill spacing in the pit area is carried out on 200 ft sections and is deemed appropriate for Mineral Resource estimation, as the spacing is sufficient to adequately define the grade of the mineralisation and the spatial grade distribution for the material left below the pit floor.

Drill core logging has been consistent throughout the years and is considered appropriate for the mineralisation style and carried out to industry standards. Drill core handling, surveying, and chain of custody from the rig to the core logging facility, was found to meet or exceed industry standards.

AMPL suggests a down-the-hole survey tool be used on some of the holes to check the extent of the deviation in future programs especially those adjacent to the known ore limits.

11.0 SAMPLING PREPARATION, ANALYSIS, AND SECURITY

A considerable effort has been made to examine the sampling and analytical procedures for the assaying of molybdenum at the Endako Property.

In 2006, CDN Resource Laboratories Ltd. #2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Phone: 604-882-8422; Fax: 604-882-8466 (www.cdnlabs.com) created a REFERENCE STANDARD: CDN-MoS-1 based on the Endako Mill Feed.

This indicates the difficulty in attaining precise and repeatable results. The method and results are given below:

- Reject ore material was dried, pulverized and then passed through a 200-mesh screen.
- The +200 material was discarded. The -200 material was mixed for 7 days in a double cone blender.
- Splits were taken and sent to 12 commercial laboratories for round robin assaying. Round robin results are displayed below (see Table 11.1 and Figure 11.1, below)

TABLE 11.1
COMPARISON OF DIFFERENT LABS

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12
	Mo %	Mo %	Mo %	Mo %	Mo %	Mo %	Mo %	Mo %	Mo %	Mo %	Mo %	Mo %
	0.059	0.061	0.066	0.061	0.067	0.068	0.068	0.063	0.068	0.069	0.068	0.061
	0.056	0.059	0.065	0.060	0.067	0.068	0.069	0.064	0.067	0.069	0.066	0.060
	0.059	0.060	0.065	0.059	0.067	0.069	0.070	0.064	0.066	0.070	0.067	0.060
	0.055	0.061	0.067	0.058	0.068	0.066	0.072	0.065	0.067	0.071	0.067	0.065
	0.060	0.059	0.064	0.062	0.068	0.066	0.073	0.065	0.067	0.070	0.066	0.062
	0.060	0.062	0.064	0.062	0.067	0.066	0.073	0.066	0.068	0.068	0.067	0.056
	0.057	0.061	0.066	0.062	0.068	0.068	0.073	0.065	0.067	0.071	0.068	0.062
	0.055	0.062	0.066	0.061	0.069	0.066	0.072	0.060	0.067	0.070	0.068	0.061
	0.057	0.060	0.065	0.064	0.067	0.065	0.073	0.064	0.068	0.071	0.067	0.063
	0.059	0.060	0.065	0.062	0.067	0.064	0.072	0.063	0.067	0.069	0.067	0.061
Mean	0.058	0.060	0.065	0.061	0.067	0.067	0.072	0.064	0.067	0.070	0.067	0.061
Std. Dev.	0.002	0.001	0.001	0.002	0.001	0.002	0.002	0.002	0.001	0.001	0.001	0.002
%RSD	3.37	1.53	1.45	2.83	0.96	2.37	2.57	2.60	0.94	1.48	1.10	3.73

Source: *CDN Resource Laboratories, 2025*

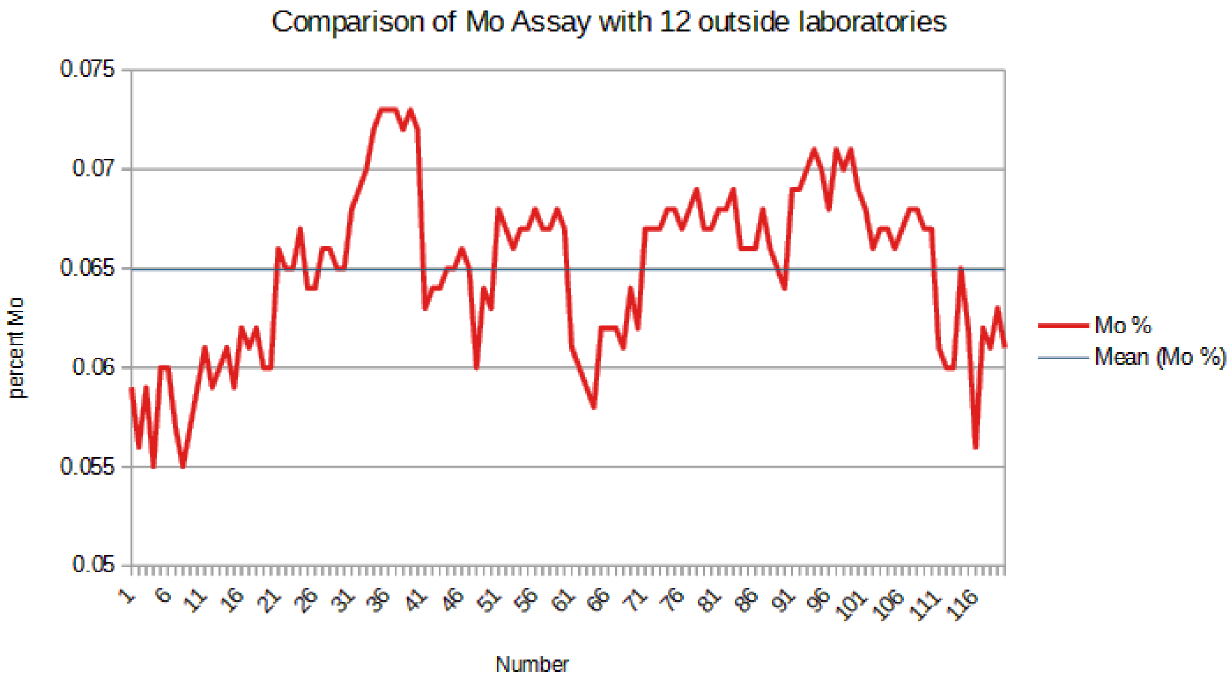


Figure 11.1 Comparison of Different Labs
***Source:** AMPL, 2025*

11.1 DISCUSSION

It is telling that in-spite of sending a single homogenized sample to 12 different laboratories, there was still considerable variance with grades ranging from a low of 0.053% Mo to a high of 0.073% Mo with an average grade of 0.065% Mo. This is based on 120 repeat assays on the same homogenised sample. This gives a range of ± 27%.

This would imply that there is a statistically significant variance in assays. It is the opinion of the QP that individual assays, by themselves, can have considerable variance in reported assays. The solution to this would be to have a larger data base to minimize the effect of single assays. As such, this would support the use of blast hole assays in resource calculations. The total number of assays will become more important than variances of individual assays.

11.2 SAMPLING METHODS AND ANALYTICAL PROCEDURES

The text from this section was sourced from the Assessment Reports that were filed with the B.C. Government as well as information obtained in IMCI report *Technical Report, Endako Molybdenum Mine*, located near Fraser Lake, British Columbia, Canada, 2011 and *WGM Technical Report on the Endako Mine*, British Columbia, Canada, Centerra Gold Inc., NI 43-101 Report, 2018. As such, most of the text has been derived from these reports and should be viewed as such.

The on-site assay lab is owned and operated by Endako Mine. It processes samples for the Endako Mine exclusively. The lab's primary purpose is analyses of blast hole samples for grade control along with samples from the process plant. The laboratory maintains its own system for quality control. Currently the lab is in a state of disrepair (see Figure 11.2, below).

Figure 11.2 Assay Lab
Source: *AMPL, 2025*

Typically, on-site laboratory procedures do not frequently change at an operating mine, but it is obvious from the text recovered from the Assessment Report, there was a change between 1991 and 2002, and possibly a second change prior to 2008. The QP cannot be certain of the exact date of the change, and therefore, in the description below, the Assessment Report reference is noted.

11.2.1 Placer Dome – Pre-1997

Sample divisions while logging core were based on regular 10 ft intervals, regardless of the lithological contacts. This procedure is common in porphyry deposits intended to be mined by an open pit extraction method. The procedure does mask grade dilution from barren dikes that are less than 10 ft wide and also dilutes the contacts at ore/waste lithological boundaries. However, considering the size of the mining equipment used at Endako, these dilution issues are not deemed material to the resource estimation.

The entire 10 ft section of drill core was brought to the laboratory in sample bags. Samples were identified by numbered tags included in the bags.

The 1991 Assessment Report described the methodology as follows:

- At the laboratory, the entire core was crushed using a jaw crusher. The crushed material was riffle split in a ¾-inch Jones splitter to obtain a 1,500 gram (g) sub-sample. The sub-sample was crushed in a cone crusher, and a 100 g sub-sample was extracted via a ⅜-inch riffle splitter. The 100 g portion was then pulverised in a ring mill pulveriser for a minimum of 2 minutes. The entire pulverised material was then bagged in a craft envelope and numbered according to the sample tag.

- Samples are analysed using a X-Met, X-ray fluorescence (XRF) spectrophotometer calibrated to the mine MoS_2 model. The instrument probe 2 was inserted in the front position of the instrument and allowed to stabilise and self-calibrate for at least 10 minutes. The pulverised samples were then transferred to Mylar bottomed analysis cups in lots of 10. A standard was run on the X-Met to ensure proper calibration. The measurement of the 0.180% MoS_2 standard was expected to assay between 0.172% MoS_2 and 0.188% MoS_2. If the instrument returned a different figure, the instrument was re-calibrated. The 10-sample lot was assayed twice, and the instrument was then re-calibrated between each sample lot.

In 1991, Endako reported a relative standard deviation of 4.5% for MoS_2 ranging between 0.02% MoS_2 and 0.350% MoS_2.

The accuracy, when compared to Atomic Absorption analysis, was as follows (see Table 11.2, below).

TABLE 11.2
ACCURACY OF THE X-MET XRF EQUIPMENT

MoS_2 Range (%)	Relative Standard Deviation
< 0.060	12.2%
0.060 – 0.110	7.5%
0.110 – 0.160	5.4%
0.160 – 0.210	5.2%
>0.210	4.9%

Source: *IMC, 2011*

The QA/QC program from the geology department did not exist during that time. The geology staff relied on the QA/QC program from the laboratory.

11.2.2 Thompson Creek Metals Company Inc. (Thompson Creek) – Post-1997

Sample divisions while logging core were mostly based on regular 10 ft intervals, regardless of the lithological contacts. This procedure was modified during the drill programs that were logged by Taiga where the samples were occasionally broken at major lithological boundaries. Core photography was also added prior to splitting.

The entire sample of drill core was brought to the laboratory in sample bags. Samples were identified by numbered tags included in the bags. The Endako Mine Laboratory is on-site and operated by the Endako employees. The Endako Mine Laboratory utilises ISO 9002 protocols, but the Endako Mine Laboratory is not an ISO/IEC Guide 43-1 or ISO/IEC Guide 25 accredited laboratory. The following describes the sampling and analytical procedures in place at the time.

11.2.2.1 Sample Preparation for Drill Core

Half core samples were crushed in a jaw crusher and riffle split to extract between 700 g to 1,000 g sub-samples. The sub-samples were oven dried, if necessary, prior to pulverising. Samples were then cone crushed and rifle split again to obtain a 75 g to 100 g sub-sample. These sub-samples were then pulverised for 2½ minutes in a ring and puck pulveriser and then placed in numerical order on a tray and transported to the assay laboratory.

11.2.2.2 Sample Preparation for Blast Holes

Each blast hole sample delivered to the assay laboratory weights 5 kg to 10 kg. The blast hole cuttings were poured into a large pan and riffle split to obtain between 700 g to 1,000 g sub-samples. The split was placed in a drying oven for approximately 2 hours. Very wet samples sometimes clump together due to clay content and occasionally had to be run through the jaw crusher a second time to break them up after drying. The sub-samples from the crusher were then cone crushed and riffled again to obtain a sub-sample weighting about 75 g to 100 g. The samples were then pulverized for 2½ minutes in a ring and puck pulveriser, which were then placed in numerical order on a tray and transported to the assay laboratory.

11.2.2.3 Analytical Procedures

The 2002, 2003 and 2006 Assessment Reports described analytical procedures for the preparation of low-grade molybdenum samples. It was mentioned in the document that these procedures applied to all samples within the range of the concentration present in the rougher tail, flotation feed and first cleaner tails. The laboratory also added that the mine blast hole cuttings and diamond drill core samples fell within this category. The procedure was described as follows.

At the assay laboratory, a 2 g sample pulp was placed in a 250 millilitre (ml) beaker. Forty ml of 30% hydrochloric acid (HCl) solution was added, and the beaker was covered. The mixture was allowed to digest at high heat for 10 minutes to 15 minutes. The solution was filtered through #2 fast fold paper into a waste catch beaker, which was washed 2 times with hot water to ensure all oxides were removed.

1. Before filtering, if the oxide content of the sample is required for analysis, the sample is to be placed in a 200 ml phosphoric flask containing 25 ml of aluminium chloride ($AlCl_3$ – Lewis acid) solution under the funnel. The sample was then washed 3 to 4 times with hot distilled water and 10 ml of HCL was added, then cooled.

2. The filter papers containing the sulphides were inserted back into the beakers and placed in front of the fuming hood. Five millilitres of HCl, 10 ml of HNO_3, and 8 ml of $HClO_4$ were added to the samples. The addition of these acids was done in this order in front of a fuming hood. Covers were put on the beakers.

3. The beakers were placed on a 3-switch plate until vigorous white fumes evolved. The beakers were left on the hot plate for 3 to 5 additional minutes and then removed and cooled. The lids and sides of the beakers were washed with distilled water and 20 ml of concentrated HCl was added. The beakers were placed on a hot plate and boiled for at least 3 minutes. The beakers were then removed from the hot plate and placed on the beaker shelf over the funnel racks in numerical order. The lids were rinsed off using distilled water in a plastic wash bottle.

4. The solution was then filtered into flasks using #2 fast fold Whatman paper. Samples were then washed 3 to 4 times with hot water and cooled to 20°C. The samples were then ready for analysis on the Atomic Absorption (AA) spectrophotometer.

5. The rougher tail and scavenger tail samples were processed slightly differently. The analytical results were recorded digitally, and the report was sent to mine engineering and geology via e-mail.

The assay procedures outlined above report total molybdenum or % Mo. These results are divided by 0.5994 to report % MoS_2 or molybdenum in molybdenite form. There is no compelling reason to work in MoS_2 as there is little, if any, observed molybdenum oxide or ferromolybdenate. The primary reason to maintain the use of MoS_2 is that it has been used traditionally at Endako, and the entire operation is accustomed to that format.

In the 2008 Assessment Report, Taiga reported the Endako Assay Laboratory was using a XRF analyser supplied by Rigaku. There were no laboratory procedures appended to the report. The 2008 QA/QC report indicated that in 2008 the XRF spectrometer was new, therefore, the WGM assumed the analytical procedure changed prior to the 2008 drill program. All geology samples were processed with this equipment with a portion re-analysed using AA as part of the laboratory QA/QC program.

11.2.3 Quality Assurance and Quality Control (QA/QC)

QA/QC programs were rare prior to 1997 and consisted mostly of re-submitting pulps to an umpire laboratory.

The earlier Endako Mine QA/QC report that was available for review was titled *QA/QC Program – Review of Results from the 2007 Exploration Drilling Program, Endako Mine Area*. The report was written by Mr. M. Jamieson, P.Geol. of Taiga and was dated August 21, 2008. Taiga was a group of Professional Geoscientists offering geological services from core logging to managing full fledge exploration programs. The company is now closed due to the retirement of its principal owners. Taiga was independent of Thompson Creek at the time the report was completed.

Information on QA/QC programs, prior to the 2007 drill campaign, is non-existent and the QP assumes that the programs in place, prior to 2008, were restricted to the laboratory's own QA/QC protocols.

The following sections summarise the available reports, with comments from AMPL.

11.2.3.1 Quality Assurance and Quality Control (QA/QC) – 2007

Taiga was contracted by Thompson Creek to review and evaluate the results of the QA/QC program implemented for the 2007 exploration diamond drill program. The original QA/QC program was designed by Mr. Philip Mudry of Taiga, in consultation with the Endako Mines personnel, and consisted of inserting control samples into the sample stream. Duplicate assays and third-party check assays were also completed. The report reviewed the analytical results of 352 control samples.

For each 100 samples submitted, 2 Blanks, 3 Standards and 5 Duplicates samples were included at regular intervals amounting to a 10% overall insertion rate. In addition to the control samples, check samples were sent to ACME Analytical Laboratories Ltd. (ACME) in Vancouver for independent analysis. These check samples consisted of a random selection of coarse rejects and pulps, selected by the Project supervisor.

11.2.3.2 Blanks

Blanks material consisted of pre-packaged 20 g craft envelopes containing pulverised silica sand. The material was purchased from WCM Minerals, a division of WCM Sales Ltd. of Burnaby, British Columbia, Canada. This material was suitable for assessing analytical contamination but not suitable for assessing cross contamination during the sample preparation stage.

Taiga reported all blanks submitted, with the exception of two, returned with low values within the four times detection limit. One of the sample batches with the elevated blank was re-run and the new results were considered correct.

11.2.3.3 Standards

Endako used four Certified Reference Material (CRM) Standards purchased from WCM Minerals, which provides reference materials (RM) with a range of low-grade, mid-grade and high-grade copper-molybdenum-silver standards. Table 11.3, below, summarises the accepted values and standard deviation for the RM.

TABLE 11.3
CERTIFIED REFERENCE MATERIALS USED DURING THE 2007 DRILL PROGRAM

Standard Name	Number Inserted	Accepted MoS_2 Value (%)[1]	Standard Deviation (MoS_2 %)[1]	Endako Assay Results
CU111	23	0.192	0.0072	Good accuracy – low bias
CU119	24	0.114	0.0036	Moderate accuracy – low bias to sample 12024, poor accuracy after that
CU120	24	0.080	0.0036	Moderate accuracy – high bias
CU124	22	0.048	0.0016	Poor accuracy – high bias

[1]Values were converted to MoS_2 % from the Mo% provided by the manufacturer

Source: *WGM, 2018*

Ninety-three (93) CRMs were inserted into the drill core sample sequence: 23 samples of CU111, 24 samples of CU119, 24 samples of CU120 and 22 samples of CU124. WGM examined the graphs provided in the report and compiled the following comments:

- For the high-grade standard CU111, only one value exceeded the three times standard deviation. No other data point showed two consecutive samples exceeding the two times standard deviation. Endako assays displayed a reasonable spread with the expected value, and the data does not suggest a systematic bias. The standard submitted to ACME shows three failures out of the four samples submitted.

- For the moderate-grade standard CU119, the data shows a degradation of precision starting at sample 12024, where the Endako analytical results displayed five failures above the ±3 standard deviation and multiple failures of two consecutive samples exceeding the two times standard deviation. This comment is only valid, if the sample numbers are consecutive from the start of the drill program. Prior to sample 12024, there were no failures, but the chart shows a slight negative bias. The degradation of precision is not apparent in the other three charts for the same range of sample numbers; therefore, WGM concludes the problem was likely related to the Standard itself and not the Endako assays.

It is noted, that out of the three ACME assays, one sample exceeded the ±3 standard deviation.

- For the moderate-grade standard CU120, the data shows three failures above the ±3 standard deviation and three occasions where two consecutive samples exceeded the two times standard deviation. The data shows a high positive bias for the Endako assays. The four samples returned by ACME showed one failure above the ±3 standard deviation and two consecutive samples exceeding the ±2 times standard deviation. One major issue is that ACME assays appear to be biased low while Endako assays indicate a bias high.

- For the low-grade Standard CU124, the data indicated a high bias for the Endako samples resulting in multiple failures above the ±3 standard deviation. The three ACME samples did not fair much better with two data points below the ±2 standard deviation and showing a possible low bias.

A total of 228 replicate (check) samples were selected from Endako's pulps (91 samples) and coarse rejects (137 samples) and sent to the ACME Laboratory for independent analyses. Unfortunately, the analytical procedure used by ACME, a multi element emission spectrometer (ICP-ES), does not mimic the AA method used by the Endako Laboratory.

The pulp duplicate graph displayed in the report indicated a slope of regression of 0.891 and an R-squared value of 0.983. This data shows a high correlation between the Endako and ACME data, but the slope of regression indicates the Endako Laboratory consistently returned a higher-grade than the ACME Laboratory. This is also visible in the CRM sample results. The variation is on average 10% greater molybdenum in the Endako samples than in the ACME samples; however, the 10% variation is independent of molybdenum content and appears consistent at all grade ranges.

The coarse reject duplicate graph displayed in the report indicated a slope of regression of 0.912 and an R-squared value of 0.973. This data shows a higher correlation between the Endako and ACME data than the pulp duplicate. The slope of regression is also better; however, the data still indicates the Endako Laboratory consistently returned a higher grade than the ACME Laboratory.

Pulp duplicates are normally better homogenised than the coarse rejects and typically performed better when compared to the Empire Laboratory assays. This trend is reversed and seems to indicate an issue related to the disparity of the analytical procedures (*i.e.*, Endako Laboratory may be digesting the sample more than ACME).

Taiga concluded that the 2007 program, as implemented, is similar to programs used by other companies in the mineral exploration sector. Control samples are being included in sufficient numbers to allow assessment of each batch of samples analysed and the CRM used covered a range of molybdenum values similar to that which is expected to be encountered in the core samples. AMPL agrees with these conclusions.

The 2007 program appears to have a good follow-up with some of the sample batches re-submitted. AMPL notes it is not clear in the report if the follow-up occurred while the drill program was in progress, or if the QA/QC data was examined only after the program was completed.

AMPL is at loss to explain why Endako used standards from other operations when a standard was developed by CDN Resource Laboratories in 2006, unless they were not aware of it at the time.

11.2.3.4 Development of an In-house Standard

Prior to the 2008 exploration program, it was determined that a set of RM would be prepared in-house, rather than purchasing standards from an external supplier. The RM derived from the Endako low-grade and high-grade ore would share the same matrix as any samples collected during diamond drilling, which would facilitate comparisons and analyses for QA/QC purposes.

Three RMs were planned; one representing low-grade ore near Endako's run-of-mill ore grade, one representing high-grade ore and one representing super high-grade ore. For each RM, half-splits from eight rejects were blended into composites to match the planned values as closely as possible. These composites were then pulverised and homogenised.

For round-robin analysis, 10 portions of each composite were sent to three different external labs (ACME, ALS Chemex and Loring Laboratories) to be analysed in duplicate. A 4-acid digestion was used, and the assays were done by ICP-OES. Endako's lab, which used an HCl-HNO_3 digestion and an AA spectrometer for assaying, was also included in the round-robin. Loring Laboratories, based in Calgary, Alberta, and ACME are certified ISO 9001:2008 facilities. ALS Chemex of Vancouver is a certified ISO/IEC 170525:2005 facility.

The low-grade RM returned an average assay of 0.037% MoS_2, which is close to the cut-off at the time the RM was developed. The final RM was pre-packaged in envelopes ready for insertion in the core sampling routine. The specifications of the RM are shown in Table 11.4, below.

Table 11.4
In-House Reference Material

Standard Name	Accepted MoS_2 Value (%)	Standard Deviation (MoS_2 %)
SHG	0.148	0.006
HG	0.099	0.005
LG	0.037	0.003

Source: *WGM, 2018*

During the development of the RMs, Endako submitted the homogenised material to the various laboratories with the WCM Minerals SRM used in the 2007 drill campaign. The Z-score plot shows the ACME and ALS Chemex laboratories displaying a slightly low bias but very little drift. The Endako Laboratory shows no bias, but a large drift is present; however, no failure above ±3-time standard deviation was recorded. The Loring Laboratory shows multiple failures above the ±3-time standard deviation (Figure 11.3, below).

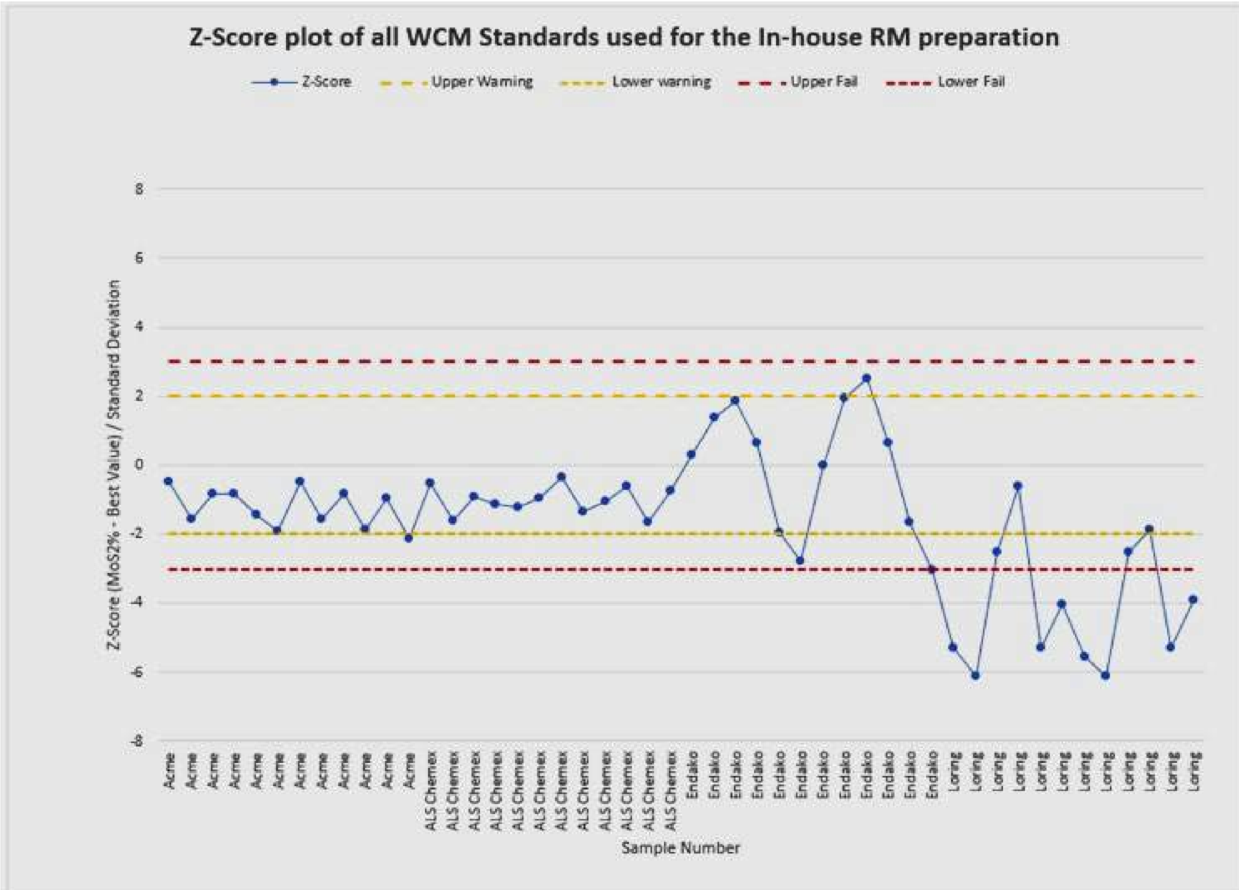


Figure 11.3 WCM Standards Submitted During the Endako In-house RM Development
Source: *WGM, January, 2018*

WGM recommended replacing the Loring assays with assays from a different laboratory such as ActLab or SGS since the Loring results would affect the standard deviation of the RM, which in turn will affect the warning and fail limits.

Again, AMPL is at loss to explain why Endako used standards from other operations when a standard was developed by CDN Resource Laboratories in 2006, unless they were not aware of it at the time. These standards were available for purchase.

11.2.4 Quality Assurance and Quality Control (QA/QC) – 2008

For the 2008 drill program, Taiga reported Endako's in-house laboratory was used as the primary lab for the assaying of all drill core sampled during the 2008 exploration program. Taiga also reported that all samples were assayed by XRF for MoS_2 content (reported in percent, with a detection limit of 0.001%). As stated earlier, it appeared the analytical procedure changed from a 3-acid digestion with AA finish to XRF shortly after the development of the RM in 2007 and before the start of the 2008 drill program. WGM could not confirm this was the case.

Ten percent (10%) of the samples submitted to the assay lab were control samples, comprised of Duplicates (5%), Standards (2.5%) and Blanks (2.5%). Sample numbers for the controls were selected at random and compiled into a list for the core splitter. Every second control sample was a duplicate and the remaining controls were standards or blanks. The Empire Laboratory changed to ALS Chemex (from ACME) for the 2008 drill program due to the better performance exhibited during the preparation of the Endako in-house RM.

Throughout the 2008 drill program, batches were re-submitted to the laboratory when a control sample failed, when sample numbers appeared to be out of order or when assays did not appear to be consistent with drill core logs. A second round of assays was done with either XRF or both XRF and AA. If a significant discrepancy was apparent between the new assays and the originals, the second set of values from the XRF was generally adopted. If there was no appreciable difference, the original values were retained.

11.2.4.1 Blanks

Two types of blanks were used during the program (not concurrently). A pulp blank obtained from WCM was used for the first 24 submissions, and that was replaced by a coarse blank for the remainder of the program. The coarse blank was a composite of barren split drill core from the granitic Casey unit located just north of the Endako Mine, assaying at or below the 0.001% detection limit for MoS_2. Taiga considered a sample exceeding four times detection as a failure. Early in the program, Taiga reported that failures and near failures were common. Eventually, the issue was resolved by tweaking the calibration of the XRF machine for samples near detection limit.

11.2.4.2 Standards

Three RMs described earlier were distributed randomly in the sample stream. A sample mix-up occurred on two occasions, and a procedure was implemented to reduce the chance of this re-occurring. For each submission, the assayed grade was checked to determine if it fell within the range of the expected standard value ±2 times the standard deviation, which is the pass/fail criterion used by Taiga. A total of 73 standards were submitted: 24 low-grade, 25 high-grade and 24 super high-grade.

Results from the program indicated that for the low-grade RM, one early failure was attributed to the use of the then new XRF spectrometer. The results for the high-grade RM were almost always near the expected value although a slight high bias was noted. The super high-grade RM saw one failure that could not be corrected by re-assaying the batch. This failure was considered as an outlier. All other samples were almost always near the expected value although a slight high bias was noted.

WGM inspected the data provided, and as stated in the Taiga report, all samples returned a very small high bias. This can be attributed to the 3-acid and 4-acid digestion used to develop the RM, which may not have completely digested the homogenised material. Failures were almost non-existent (see Figure 11.4, below).

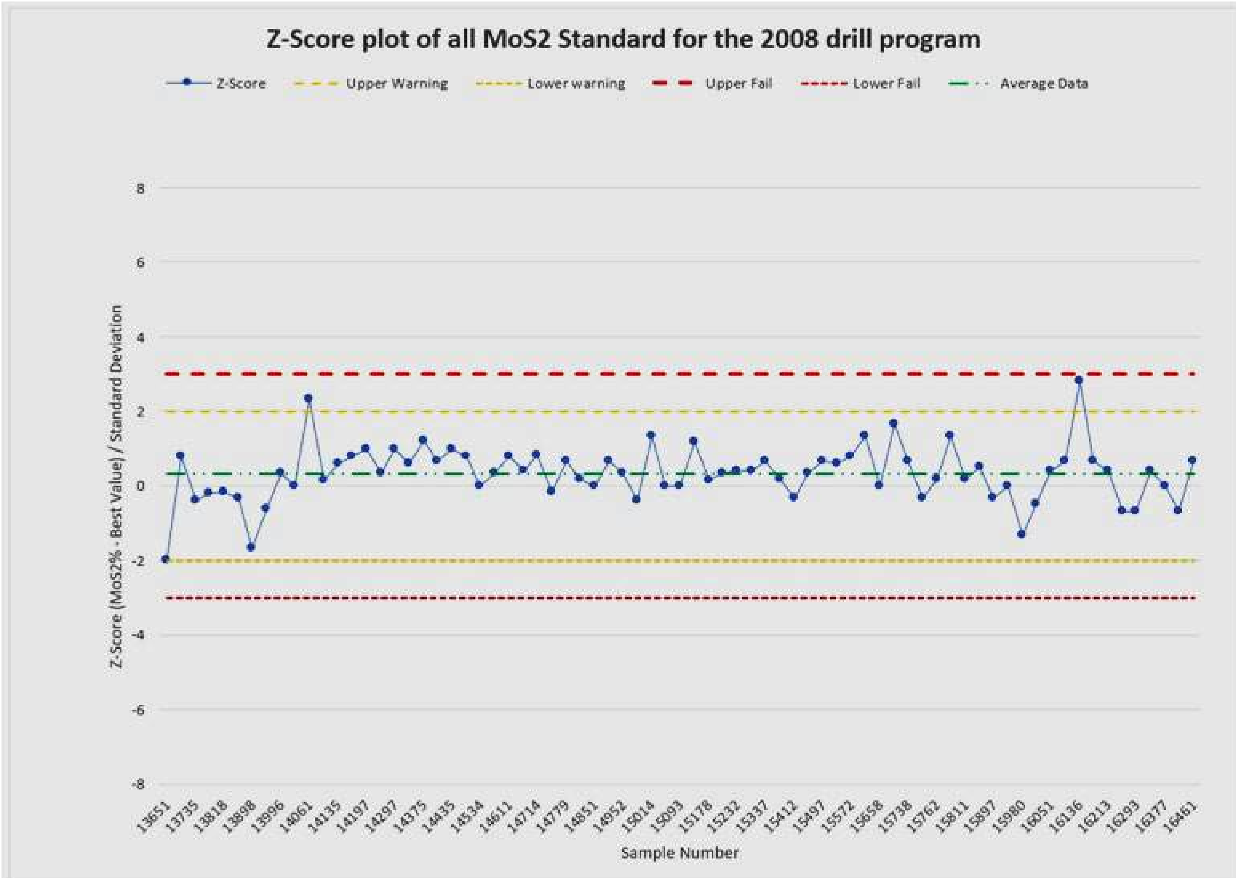


Figure 11.4 Z-Score Chart for 2008 In-house RM Results
Source: *WGM, January 2018*

11.2.4.3 Coarse Duplicate

A total of 138 duplicate samples were submitted for analyses during the 2007 drill program. Duplicate samples consisted of ¼ core samples, which typically exhibit more variability due to the smaller volume when compared to the ½ core split of the original samples.

No chart was presented in the report; therefore, WGM graphed the data that was available. The chart indicated significantly more scattered than the 2007 quarter core duplicate. The slope of regression was 0.7882 and the R-squared value was a poor 0.667, suggesting a bias existed between the first half of the core when compared with the second ¼ sample. WGM noted that a few high-grade samples negatively influenced the regression. The data was filtered to eliminate assays above 0.1% MoS_2, the new regression indicated a 0.935 slope and an R-squared of 0.628 indicating reasonable spread about the parity line but with large variation between the pairs.

11.2.4.4 Replicate Sample to Empire Laboratory

A total of 126 of samples was submitted as both pulps and rejects to ALS Chemex for analysis by XRF (procedure code Mo-XRF05) and the remaining 145 samples were sent as pulps only. The submission rate amounted to 10% of the core samples analysed.

Rejects were riffled and pulverised at the ALS Laboratory prior to analysis. One set of pulps prepared at Endako was re-pulverised at ALS to meet their QA/QC specifications. This was mentioned to Endako's chief assayer, who then increased the time future samples spent in the pulveriser.

Taiga reported that ALS returned values that were generally very close to those given by the Endako Laboratory, particularly when the labs assayed the same set of pulps.

WGM examined the chart provided and found that for pulps with a linear regression showed a slope of regression of 0.944 and a R-squared value of 0.99. For the rejects, the linear regression showed a slope of regression of 1.060 and a R-squared of 0.976. The data suggest excellent reproducibility of the Endako assays.

11.2.5 Quality Assurance and Quality Control (QA/QC) for the 2010 Drill Program

During the 2010 drill campaign, the QA/QC program implemented in 2008 continued. A dedicated QA/QC report covering that period was not available for review; however, Mr. J. Marek, P.E., of IMC commented on the result of the QA/QC program for 2010 in the technical report titled T*echnical Report Endako Molybdenum Mine* dated September 12, 2011. Of all samples submitted to the assay lab, the control samples insertion rate increased to 15% during the 2010 drill program.

11.2.5.1 Blanks

There were 251 blanks inserted in 2010. The material was the same material as the 2008 program. The highest reported value was 0.007% MoS_2. Only three were above 0.005% MoS_2, indicating improvement over the 2007 and 2008 results.

11.2.5.2 Standards

Endako continued using the in-house RM for 2010. These were inserted at a rate of 3.5 out 100. IMC reported that all RM results were acceptable.

11.2.5.3 Coarse Duplicates

Duplicate samples consisted of ¼ core samples for 2010. A total of 404 samples were submitted. IMC reported the results in all years (2007 to 2010) indicated the duplicates are not biased relative to the initial samples. The T-Statistics (2 Tail) indicate the original and duplicate sets can be assumed to be from the same populations with more than 95% confidence. It was also noted a reasonable amount of variation existed in the sample results, which indicated the normal variability in sample preparation and assay procedures. WGM does not agree with this statement. The variability is likely resulting from the smaller sample size (volume variance issue) and the inherent nugget effect.

11.2.5.4 Replicated Samples to Empire Laboratory

Check assays have been sent to outside third-party laboratories for 2010 on a 1 in 10 basis. During 2010, the Empire Laboratory remained the same as 2008 and both pulps and rejects were sent to ALS Chemex for analysis by XRF.

IMC reported that all data sets did not indicate any measurable bias in the check results and the data sets indicated the check assays can be assumed to be from the same populations, with more than 95% confidence.

WGM examined the graph provided in the report and the data shows an even spread about the parity line and no systematic bias was visible.

11.2.6 Quality Assurance and Quality Control (QA/QC) – 2011 and 2012 Drill Programs

According to the Assessment Reports, during the 2011 drill programs, QA/QC measures were implemented throughout the program both by the core shack sampling procedures and by the Endako Assay Laboratory. Approximately 15% of the samples were QA/QC control samples consisting of Blanks (<0.002% MoS_2, detection), low-grade (0.037% MoS_2), high-grade (0.099% MoS_2) and super high-grade (0.148% MoS_2) RM. Additionally, sample duplicates of reject and pulps were completed to an Empire Laboratory.

A QA/QC report covering that period is not available for review. It is, therefore, unclear as to the follow-up of the program. The QP was unable to retrieve the assay results for the control samples although from examination of the assay sequence in the logs and comparison of that to the assay sequence in the assay certificate, it is obvious that control samples were indeed inserted.

11.2.6.1 New Drill Holes Versus Historic Drill Holes

The text from this sub-section was sourced and summarised from the technical report titled *Technical Report Endako Molybdenum Mine* authored by Mr. Marek, P.E., of IMC and dated September 12, 2011. Comments from WGM are inserted when appropriate.

The QA/QC information, developed by Endako since 2007, provides comfort that the diamond drill results are reliable for development of a resource model, mine plan and corresponding Mineral Reserves and Mineral Resources. Most of the diamond drilling on the Project is historic in nature and WGM is aware that no comprehensive QA/QC program existed prior to 2007.

In order to ensure the historic drill holes are usable in the resource estimate, IMC compared closely spaced new drilling (2007-2008) with historic drill hole data. The data from the 2010 drill program was omitted because the holes were drilled in the Denak Northwest Extension and they were mostly surrounded by newer holes drilled after 2008.

The drill data was composited in 44 ft intervals and grade was length weight averaged within the composite intervals. The 2007-2008 composites were identified and the closest distance from the mid-point of the 2007-2008 composites to the surrounding historical drill hole composites were calculated. IMC compared the paired data at a range of different spacings. In addition to graphing the data, a series of statistical hypothesis tests were developed at alternative data separation distances.

Table 11.5, below, which was sourced from IMC's work, summarises the results at data spacing of 50 ft and 100 ft. The 50 ft spacing corresponds to the size of one reserve block in the Endako Mine model.

TABLE 11.5
NEAREST NEIGHBOUR COMPARISON OF 2007-2008 DRILL DATA VERSUS HISTORIC DRILL DATA

Nearest Neighbor Comparison of 2007-2008 Diamond Drilling versus Historic Diamond Drilling									
Maximum	Number	2007-2008 DDH		Earlier DDH		Hypothesis Tests at 95% Confidence			
Separation Feet	of Pairs	Mean %MoS2	Variance	Mean %MoS2	Variance	T-Test on Means	Paired T Test	Binomal Test	Komologorov Smirnoff
50	26	0.045	0.0014	0.043	0.0022	Pass	Pass	Pass	Pass
100	107	0.054	0.0028	0.049	0.0019	Pass	Pass	Pass	Pass

Source: *J. Marek, IMC, 2012*

The X-Y plots shown on Figure 11.5, below, were extracted from the IMC report. The data indicated substantial variability. IMC reported the hypothesis tests, and the quantile-quantile (QQ) plot indicated closely spaced diamond drill hole data, represents the same population independent of the time frame when the diamond holes were drilled.

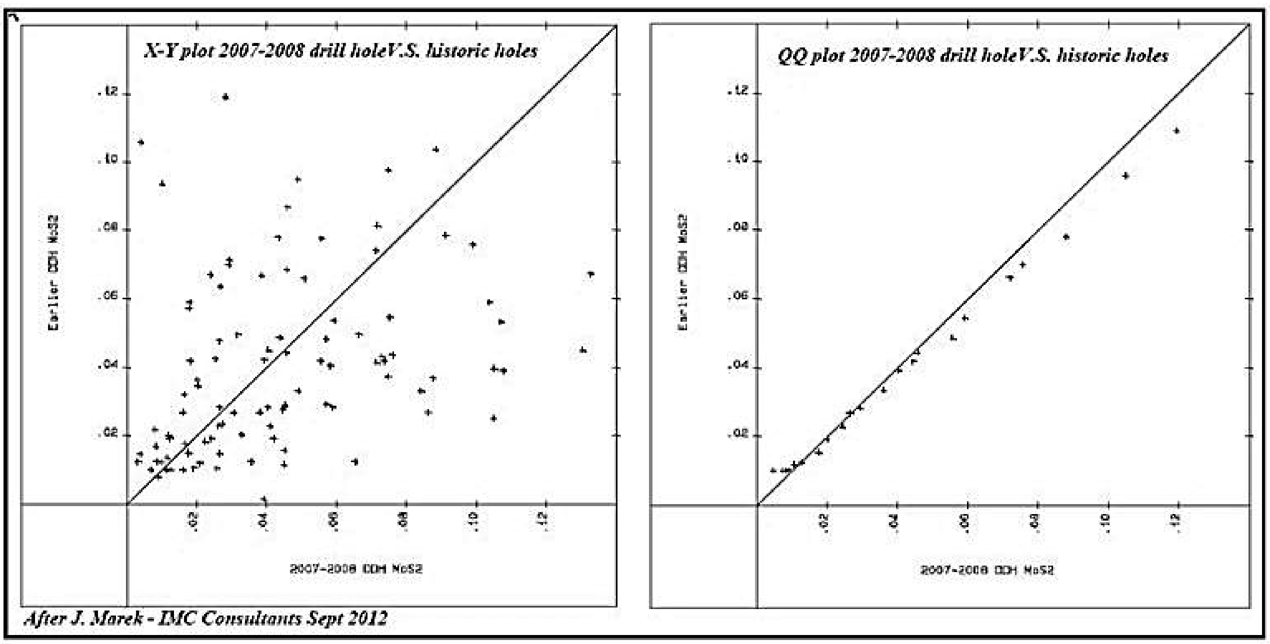


Figure 11.5 X-Y and QQ Plots – New Drill Holes Versus Historical Holes
Source: *WGM, January 2018*

WGM comments that despite the scattered nature of the data, the X-Y plot does not seem to display a bias. The QQ plot indicates that above 0.06% MoS_2 the new drill holes tend to be higher-grade than the surrounding historical holes. It can, therefore, be concluded that mixing historical holes with the newer holes will only introduce some conservatism to the resource model at the higher cut-offs.

11.2.7 Diamond Drill Holes Versus Blast Holes

The text from this sub-section was sourced and summarised from the technical report titled *Technical Report Endako Molybdenum Mine* authored by Mr. Marek, P.E., of IMC and dated September 12, 2011. Comments from WGM are inserted when appropriate.

Previous modelling work at Endako has made use of blast hole data during the model assembly process and the use of both data sets remain for the resource estimate described in Section 14.0 of this report. In order to use the blast holes and drill holes together, the data sets need to be compared to ensure that when combined together, one or the other will not introduce a significant bias in the estimate. The comparison is also necessary because the blast holes do not have the same level of QA/QC as the drill hole data. IMC used a Nearest Neighbor Comparison to gain confidence in the utilisation of the blast hole data.

IMC reported that all the comparisons pass the hypothesis tests for representation of the same population, with the exception of the 20 ft data spacing at Denak, which just barely fails the Komologorov Smirnoff population test.

IMC reported that within Endako, the diamond drill hole mean values were about 3% higher than the mean grade of the nearby blast holes. Within Denak, the 10 ft and 20 ft spacing tests reverse the trend between them. With 20 ft spacing, the blast hole values are about 7% higher than the diamond drilling data.

IMC reported that production history for 2008 and 2009, which was primarily from the Denak area, indicated the blast hole information agreed well with the reported mill head grades. IMC also reported that within the same volume, the diamond drill hole average grade was lower than the mill reported head by as much as 14%. It is the opinion of AMPL that this may be in part due to the truncating of assays in the database. It is not known when this truncating occurred, *i.e.*, before or after the original resource calculation. Nonetheless, some of the assays were unreported at least at some time.

IMC concluded that although the hypothesis tests indicate the blast hole and diamond drill data can be co-mingled, the low values of diamond drill data in Denak pose some issues for the proper estimation of head grades in Denak. WGM commented that higher-grade blast hole data will likely have a greater effect in the Mineral Resource blocks estimated in proximity to the Denak Pit bottom and affect the local estimated grade and not the global figures. In addition, AMPL is of the opinion that correcting errors in data entry may rectify this apparent error.

11.3 QP COMMENTS ON THE QA/QC PROGRAM

The QA/QC program at Endako Mine was implemented for the 2007 drill campaign. As with any new program, there were many issues affecting the results of the early program.

Taiga concluded in 2007 that the QA/QC protocols, as implemented, were to industry standard. Control samples were being included in sufficient numbers to allow assessment of each batch of samples analysed and the CRMs used covered a range of molybdenum values like that which is expected to be encountered in the core samples. WGM agrees with these conclusions.

Subsequent QA/QC data collected in 2008 and beyond indicated excellent precision and accuracy. Analysis at an Empire Laboratory indicated good reproducibility of the data generated by the Endako Assay Laboratory.

Of the data collected by Endako, prior to 2007, which did not have a QA/QC program in place at the time the holes were drilled, IMC compared the nearest neighbour assays. WGM reviewed the data available and comments that despite the wide scatter, the X-Y plot does not seem to display a bias. The QQ plot indicates that above the 0.06% MoS_2, the new drill holes tend to be higher-grade than the surrounding historical holes. WGM's opinion is that mixing historical holes with the newer holes will only introduce some conservatism to the resource model at the higher cut-offs.

AMPL is of the opinion that the QA/QC program utilized at Endako is adequate for the calculation of resources. Of note, it would appear that individual variability of assays, as noted in the pre-amble, may have greater impact on the reporting of grades. The probable solution is the use of more available assays (as in blast holes) rather than relying on individual assays.

12.0 DATA VERIFICATION – QA/QC

Field inspection and data validation, by an independent consultant, was completed in 2011 by Mr. J.M. Marek., P.E. of IMC as part of the Mineral Resource Estimate of the Endako deposit dated September 12, 2011. The data validation performed by IMC focussed on QA/QC procedures, the comparison of new drill holes against the historic holes and the comparison of the blast hole assays against the drill hole assays. A summary of the last two items, performed by IMC, can be reviewed in Section 11.0 of this report. A subsequent field inspection and data validation assessment was undertaken in 2017 by Mr. P. Desautels, P.Geo. of Watts Griffis and McQuat (WGM).

This section presents portions of Mr. Desautels' text (reviewed and accepted by Mr. Bakker Finley of AMPL), as prepared for the unpublished Centerra Gold Feasibility Study 2018.

12.1 WGM FIELD INSPECTION – AUGUST 2017

Mr. Desautels, P.Geo., visited the property August 23-24, 2017. His observations follow.

No drilling program was in progress during the WGM site visit, as the Endako Mine was on care and maintenance status since December 2014.

For the drill holes inspected, WGM found the core was properly marked. Sampling intervals were approximately 10 ft in length. The NQ sized core is split longitudinally with a manual splitter. Since the mineralisation at Endako occurs in stockworks of thin veins, veinlets and mineralised fractures, the location of the split line location is not a concern. The core was logged in the core logging facility located at the Endako Mine and all assays were performed at the Endako Mine Laboratory. Core for the most recent drilling (post-2006) is available for review and is stored on-site in cross stacked piles located on the low-grade stockpiles. Core rejects were stored in sea containers and in sealed drums at the same location, while pulps were stored in the core logging facility.

During the 2017 site visit, WGM collected five-character samples. WGM personally packaged the samples, which were subsequently shipped, via Canada Post, directly to Activation Laboratories Ltd. (ActLabs) at 41 Bittern Street, Ancaster, Ontario, Canada. The sample analysis was intended to allow an independent laboratory to assess differences in terms of grade ranges. Samples were analysed for molybdenum using procedure code "8", which is described as a 4-acid total digestion followed with inductively coupled plasma mass spectrometry (ICP-MS). The digestion uses nitric, perchloric, hydrofluoric and hydrochloric acids. This method differs from the laboratory procedure at the Endako Laboratory, which, since 2008, uses an XRF analyser with a detection limit of 0.001% MoS_2. Prior to the 2008 drill program, the Endako Laboratory used a 2 g pulp treated with a solution of potassium chlorate, followed by hydrochloric acid digest and aluminium tri-chloride treatment, followed by analysis via AA. The character samples at ActLabs were also analysed for bismuth (Bi), cadmium (Cd), cobalt (Co), copper (Cu), nickel (Ni), lead (Pb), selenium (Se), tin (Sn), titanium (Ti), uranium(U) and zinc (Zn).

The values obtained by the independent laboratory correlates well with the analytical results from Endako for the two ½ core duplicates. For the pulp samples, the Endako Laboratory produced a higher grade on two of the samples (Table 12.1, below). The higher-grade of the Endako Laboratory results could be attributed to the difference in analytical procedures.

TABLE 12.1
INDEPENDENT CHARACTER SAMPLE RESULTS VERSUS ENDAKO

Hole-ID	Interval		WGM Sample Result		Endako Sample Result		Sample Type
	From (ft)	To (ft)	Sample Number	Mo (%)	Sample Number	Mo (%)	
11B-008	577	587	83661	0.0313	107501	0.031	1/2 core duplicate
12-002	222	227	83662	0.0029	114517	0.003	1/2 core duplicate
11A-45	237	247	83663	0.001	109586	0.011	Pulp
11A-013	107	117	83664	0.002	113344	0.008	Pulp
11A-062	332	337	83655	0.002	107384	0.002	Pulp

Source: *WGM, 2018*

Assay results on the check samples indicated no other anomalous elements.

Down-the-hole surveys are rare in the Endako database. Routine down-the-hole surveys were introduced during the 2008 drill campaign. In 2008, Endako used a Flexit™ survey instrument on non-vertical drill holes. The instrument type used during the 2010 and 2011 drill program is a PeeWee™ survey instrument supplied by Equipment Jexplor Inc. and was used in the non-vertical drill holes. Prior to 2008, Endako occasionally used acid tests, especially during the 2006 drill campaign, and a Sperry Sun™ instrument was used during the 1989 drill campaign on inclined holes.

No recent bulk density measurements are available. Endako relied on the historical bulk density of 2.563 tonnes per cubic meter (t/cm^3), which is derived from production history. On the logs, density measurements were collected on every 10 ft sample. This practice was discontinued sometime between 1979 and 1980.

AMPL compiled density data on a sub-set of the drill logs. A total of 1,225 density values were extracted from 26 holes. Results of this study indicated density ranging from 2.55 t/m^3 to 2.63 t/m^3 with an average of 2.57 t/m^3 (Table 12.2, below). The methodology used to measure the density of the drill core is unknown.

TABLE 12.2 TABLE OF SPECIFIC GRAVITY			
Hole ID	**Count**	**Average Density**	**Zone**
S341	47	2.56	Denak
S342	60	2.55	Denak
S346	61	2.56	Denak
S347	58	2.57	Denak
S348	47	2.55	Denak
S349	58	2.56	Denak
S350	69	2.55	Denak
S351	38	2.56	Denak
S354	56	2.56	Denak
S355	49	2.56	Denak
S356	56	2.57	Denak
S357	56	2.55	Denak
S358	60	2.57	Denak
S359	38	2.57	Denak
S412	41	2.58	Denak
S414	34	2.55	Denak
S415	44	2.61	Denak
S416	27	2.60	Denak
S462	5	2.59	Denak
S476	75	2.60	Denak
S541	28	2.63	Denak
S551	27	2.65	Denak
S384	50	2.58	Endako
S387	49	2.57	Endako
S405	51	2.56	Endako
S409	41	2.63	Endako
Total	**1,225**		
Median		**2.57**	

Source: *AMPL, 2025*

The in-situ rock density of 2.563 t/m^3 used in the resource model is 0.3% lower than the average density compiled from the sub-set of the logs and for all intents and purposes identical. AMPL elected to keep using the density as reported by Endako.

WGM states that a comprehensive QA/QC program was implemented during the 2007 drill campaign and continued through to the end of the 2011 drill campaign. During the 2008 drill program, the sample insertion rate was 10%; the insertion rate was upgraded to 15% in the subsequent programs to reach the ideal 5% insertion rate for each of the control samples. Samples consist of blanks, standards and rejects and pulp duplicates. During the 2007 drill campaign, Endako used purchased standards from the WCM Laboratory. For the 2007 drill program, the Endako assay results for the standard indicated poor precision with multiple assays returning value above three times standard deviation. Endako suspected a matrix match issue and a difference in the analytical procedure as a contributing factor. Prior to the 2008 drill program, Endako developed a set of 3 RM. The round-robin program involved in the creation of the RM utilised four laboratories (ACME, ALS Chemex, Loring and Endako). The round-robin data results were reviewed by Mr. Desautel and found to be adequate; however, the number of laboratories used was low when compared to standards obtained through a commercial laboratory, which typically used 10 laboratories. Also, the Loring Laboratory returned poor results when compared to the WCM standards. WGM suggested that the

Loring Laboratory be dropped off the round-robin and a new laboratory be selected to replace it. Endako started using 3 RM that were developed in-house during the 2008 drill program. Endako also switched from analysing samples with wet chemistry with AA to using XRF. The exact date of the change is not known but it was prior to the 2008 drill program.

12.2 DATABASE VALIDATION

Following the site visit and prior to the Mineral Resource estimation, WGM carried out an internal validation of the drill holes in the Endako database.

Following the site visit in conjunction with the Mineral Resource estimation, AMPL carried out a second site visit to conduct a further validation of the drill holes in the Endako database.

12.2.1 Collar Coordinate Validation

WGM reported that for the most recent drill holes, the collar was surveyed using a high precision Trible™ instrument once completed. Earlier holes were surveyed using a theodolite by the mine survey department. This was verified by Mr. Mike Pond, Chief Geologist, retired.

During the WGM site visit, drill collars could not be validated in the field due to access. Sixteen drill hole collar coordinates were checked against the value entered in the paper logs. There were some minor variations of less than 1 ft in the easting, northing and elevation for all the holes examined except for S-461, which showed a difference of 60 ft. In addition to this random check, the 2013 LiDAR™ photograph was georeferenced in GEMS™, and the location of the database drill collars were compared to the drill set-up and road network clearly visible in the photo. While this method is not considered a replacement for a field survey, it does allow an increased level of confidence in the location of the holes in the database and WGM stated that they were satisfied that the holes were properly surveyed.

AMPL repeated validation of drill hole collars by plotting the drill holes relative to the “pre-mining surface”. While there were some minor discrepancies, none were considered sufficient to cause any concern (see Figure 12.1 and Figure 12.2, below.

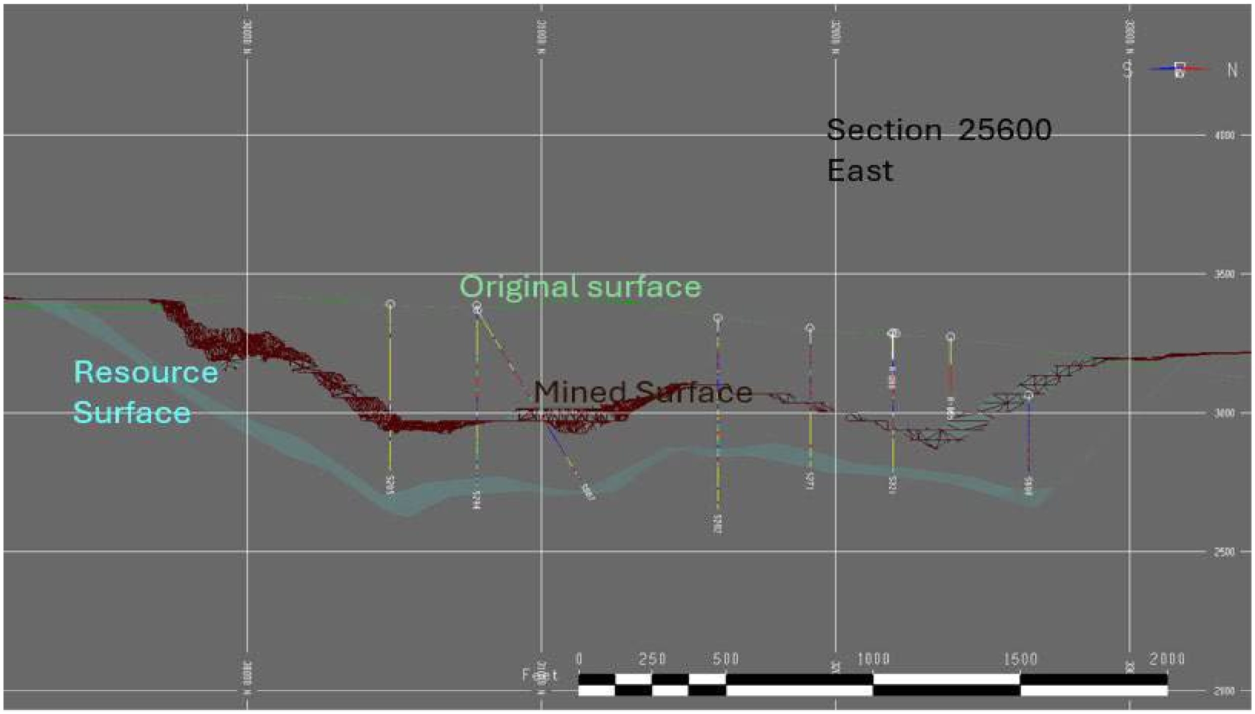


Figure 12.1 Location of Drill Hole Collars Relative to Historical Surface

***Source:** AMPL, 2025*

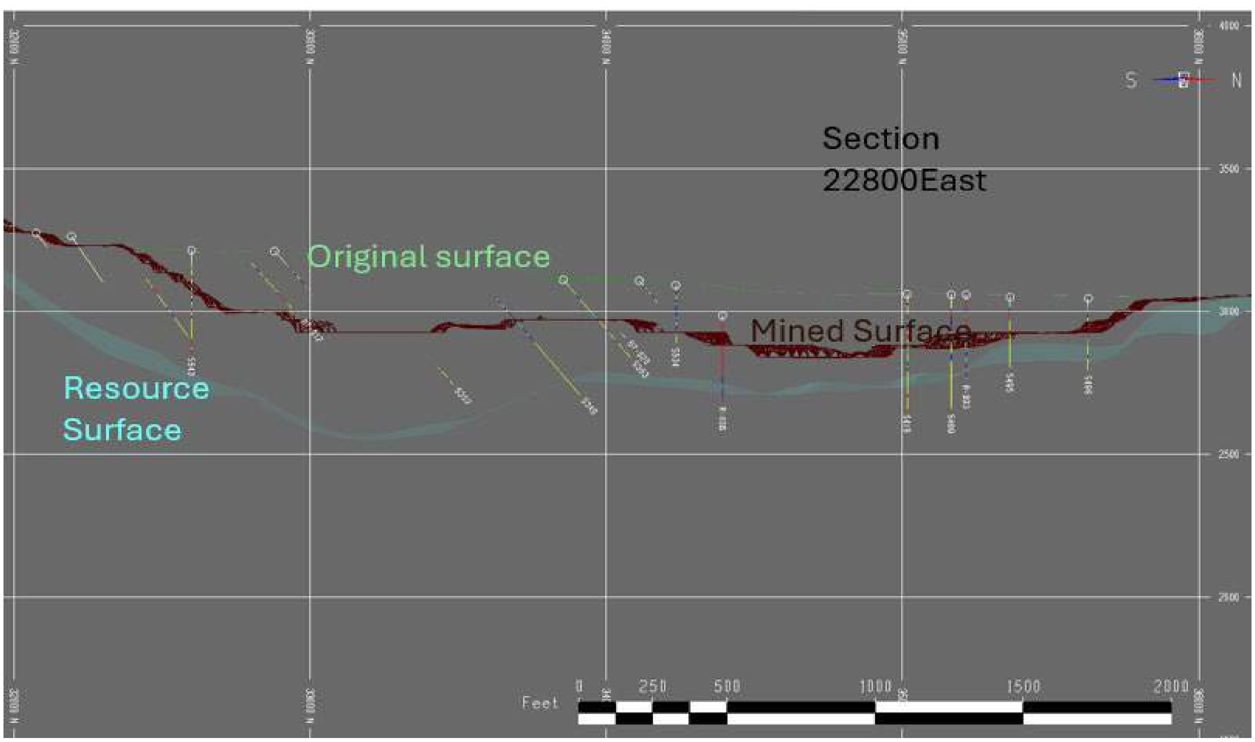


Figure 12.2 Location of Drill Hole Collars Relative to Historical Surface

***Source:** AMPL, 2025*

12.2.2 Downhole Survey Data

WGM stated that holes with downhole survey data were extracted out of the database and each deviation entry was compared with the previous one. Out of the 473 entries, one azimuth in Hole 11B-007 was obviously erroneous and three other azimuths were found to be suspicious, but not impossible. There was no obvious erroneous entry for the dip noted on the holes inspected. The erroneous entry was deleted from the database.

Following the resource estimation, WGM discovered that a number of down-the-hole survey results from holes drilled prior to 2008 were not added to the Endako Mine database. A total of 29 drill holes out of 1,317 (2.2%) had missing data. Out of the 29 holes, 20 of those holes were drilled in 2006 and surveyed with acid tests, which can easily be ±5 degrees off due to the difficulty in reading the tests accurately. The remaining nine holes are in the Placer Dome "S" series of holes drilled in 1989, which were surveyed using the Sperry Sun instrument. Five of those holes are more or less straight. The remaining four holes (S-657, S-673, S-682 and S-696), showed a deviation reaching up to 7 degrees and were collared in the Denak East Pit. This issue is not considered material to the Resource estimate since the 2006 holes affected are short and the deviation does not exceed 7 degrees at the toe of the holes. For the four holes drilled in 1998, the deviation is not considered material, especially in a porphyry deposit where a slight hole deviation issue is not considered critical. WGM recommended incorporating the missing data for completeness prior to the next Resource update. At the time of the current Resource update, it has not yet been completed.

12.2.3 WGM Assay Certificate Validation Prior to the Resource Estimate

For the historical "S" series holes, WGM validated the MoS_2 assays against the information recorded in the drill logs. For the more recent drilling and the production blast holes, WGM validated the MoS_2 assays against the Adobe Acrobat™ Portable Document Format (PDF) version of the Endako laboratory certificates. Twenty-five historical hole logs were selected at random and the MoS_2 assays in the logs were compared with the GEMS™ database entry. For the production blast holes, the MoS_2 % values on many laboratory certificates were manually compiled and these values were compared to the GEMS™ database entries. The selection of the laboratory certificates was random. Overall, WGM validated 2,486 drill hole assays out of a total of 59,497 assays. The validation rate for the drill database is 4.2%. For the production blast holes, WGM validated 498 assays out of a total of 32,927 assays for a validation rate equal to 1.5%. The archived blast hole assays were not validated but have very little influence on the Resource below the 2014 rock surface. The error rate was well below 1% for all data types and for that reason, there was no need to increase the validation rate (Table 12.3, below). Note that the erroneous samples were not corrected as they were deemed not material to the outcome of the Resource estimate.

Table 12.3
Assay Validation

Data Type	Total Validated	Number of Errors	Error Rate
Historical "S" Series Holes	1,134	5	0.44%
New Drill Holes (2006-2010)	1,352	5	0.37%
Production Blast Holes	489	1	0.20%
Archive Blast Holes	0	N/A	N/A

Source: *WGM, 2018*

12.2.4 Conclusions

WGM concluded:

> *there is no material issue related to sampling and assaying that was identified during the review of the drill data and accompanying assays. The QP finds the data collected by Endako adequately represents the style of mineralisation present on the property without a restriction on resource classification. The error rate in the drill database, for the data that was validated by the QP, was found to be very low. The Endako database exhibits good consistency for the data that was collected by the geology department since the beginning of the Placer Dome drilling and WGM is of the opinion that the current database is appropriate for resource estimation.*

The final test on whether QA/QC procedures and sampling were adequate will be in the reconciliation of what the theoretical calculated grade of the deposit is when compared to actual mill heads.

This calculation was undertaken in Section 14.0 and is repeated here.

AMPL utilised the "original topographic" surface, as supplied by Endako, to compare reported tonnes and grade to those calculated by HxGN Mine Plan™.

TABLE 12.4
ENDAKO PRODUCTION RECORD COMPARED TO CALCULATED BY HXGN MINE PLAN™

	HxGN Mine Plan™ 3D		Endako		Difference	
	Tonnes	**% MoS_2**	**Tonnes**	**% MoS_2**	**Tonnes**	
Mined	707,937,000	0.081	666,258,000	Not Reported	(41,679,000)	-6%
Milled	334,223,000	0.136	391,795,000	0.136	57,572,000	15%

Source: *AMPL, 2025*

While this does not necessarily validate the QA/QC and the accuracy of the database, it does appear to confirm that that location of the drill holes and the accompanying assay are valid. AMPL is of the opinion that the database and QA/QC protocols are acceptable for Resource calculations.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 HISTORICAL TEST WORK AND OPERATIONS

The Endako Mine and mill have been in operation since 1965 and metallurgical processes are well established. No historical test work reports were retrieved or made available for review, but production reports were made available, and provide a basis for future metallurgical performance. The material to be processed is from the same orebody historically mined. Historical operation records and performance from 2000 to 2005 are available from the 2007 Feasibility Design report (Table 13.1, below) by Hatch. These values are not used for establishing performance but included for information. More recent operations data is available and described below.

TABLE 13.1
ENDAKO MILL HISTORICAL PRODUCTION

Mill Production	2000/01	2001/02	2002/03	2003/04	2004/05
Tonnes (000)	9,386	9,641	9,706	9,350	9,604
Grade (% MoS_2)	0.128	0.122	0.111	0.112	0.099
Recovery (%)	78.16	75.71	80.04	79.2	77.2

Source: *Hatch, 2022*

Additional operations data, from 2006 to 2011, available from operations reports, are summarised in Table 13.2, below.

TABLE 13.2
ENDAKO MILL – 2006 TO 2011 PRODUCTION

Mill Production	2006	2007	2008	2009	2010	2011
Tonnes (000s)	2,342	9,942	10,768	9,759	10,176	10,652
Grade (% MoS_2)	0.104	0.100	0.116	0.098	0.095	0.092
Recovery (%)	77.04	71.75	77.26	78.30	69.93	73.16

Source: *Hatch, 2022*

13.2 CURRENT MILL OPERATIONS AND METALLURGICAL RECOVERY

A new mill was commissioned to process the Endako ore in 2012. Operations data from the new mill are used as the basis for the metallurgical recovery predictions.

The period of data selected is from May 2013 to December 2014, inclusive. This period represented a steady state of operation before the shutdown in 2015. Data from 2012 through to May 2013 was not used as it represented the ramp up and learning period for the operation of the new mill.

Of the 601 operating days available in the May 2013 to December 2014 period, additional data points are excluded to remove issues related to plant availability, outlier feed grades and outlier tailings grades, leaving 457 data points as the basis of the metallurgical recovery prediction.

Operating within the capabilities of the mill (achieving the target 80% passing 200 µm grind and staying within the mill hydraulic capacity) recovery can be related to feed grade, as shown in Figure 13.1. below. In 2014, the first full throughput operating year at Endako, 17.1 million tonnes of ore was processed, for an average rate of 46,951 tonnes per day. The average processing rate was limited by ore availability, as reported by the operations staff. Based on daily milling reports, the milling circuit was able to meet and exceed its 52,000 tonnes per day nominal design, peaking at 73.7 kilo tonnes per day. An average of 52 kilo tonnes per day was achieved on the best 315 days in 2014. Considering this performance, it is expected the mill will be capable of 52 kilo tonnes per day (56.5 kilo tonnes per day at 92% annual availability) nominal operation when ore availability issues are resolved. This is being addressed in the mining by upgrading of the mining truck fleet.

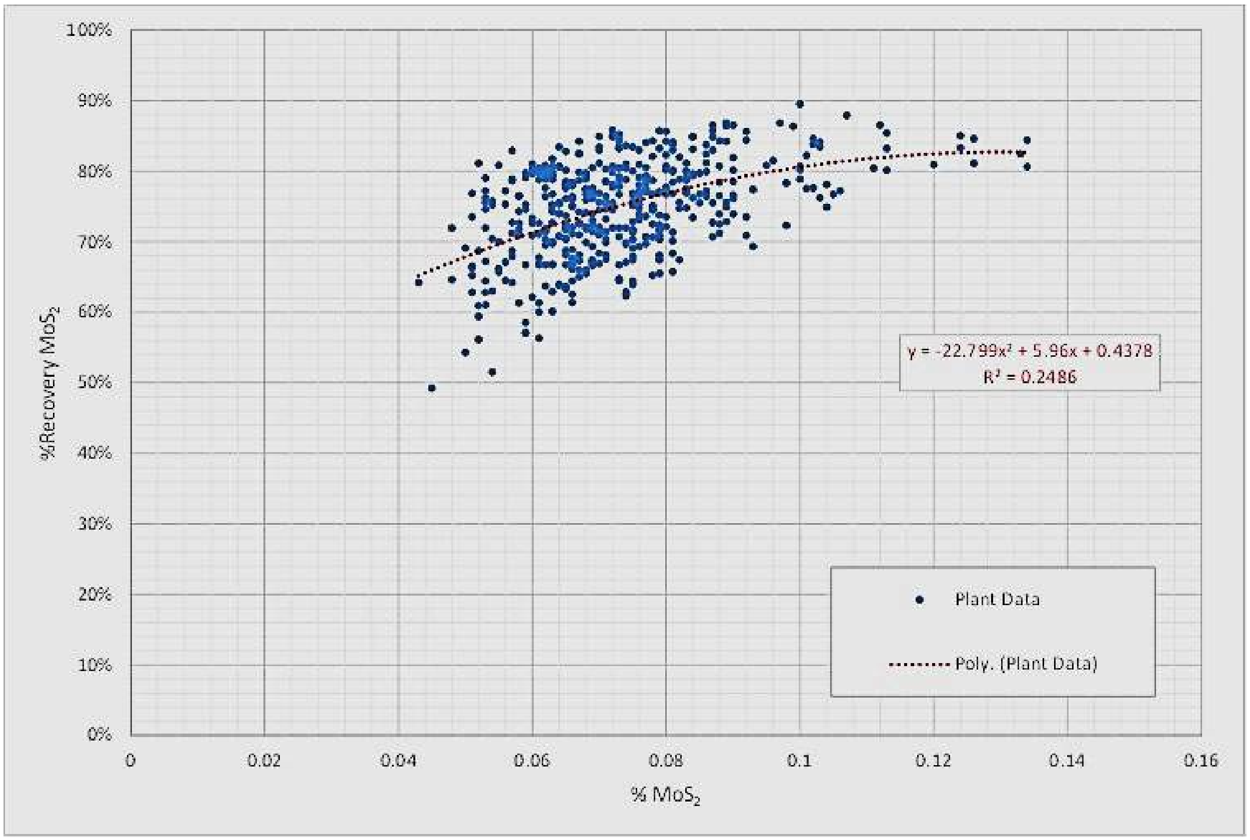


Figure 13.1 Endako MoS_2 Recovery versus Feed Grade

***Source:** Amec Foster Wheeler, 2018. Based on 2013-2014 Endako Daily Operations Reports*

The MoS_2 recovery is related to MoS_2 feed grade by the relation:

$$MoS_2 \text{ recovery} = -22.799x^2 + 5.96x + 0.4378$$

where x = % MoS_2 in feed

At higher feed grades, the recovery appears to plateau and should be capped at 82.7% for all feed grades greater than 0.134% MoS_2 (the data set peak grade). Similarly, the lowest grade at which the recovery curve is applicable is 0.04% MoS_2.

In the entire May 2013 to December 2014 period, the Endako mill produced concentrate averaging 89.7% MoS_2. In the 2014 operating year, average concentrate grade increased to 90.6% MoS_2. The concentrate produced was clean and saleable, with no significant deleterious elements.

13.3 METALLURGY REVIEW

No new process flowsheet metallurgical test programs for the existing processing plant are planned for the Endako Restart Project. Hatch (2022) reviewed the historical test work and available plant data from the 2014 Endako Expansion Project to confirm the process design criteria for the Endako Restart Project.

13.4 COMMINUTION

The Endako Restart Project is based on a primary processing capacity of 52,000 tonnes per day (or 18.8 million tonnes per annum). This will be accomplished with the existing grinding line to process the Endako potentially economic mineralisation.

The grindability test work conducted on the Endako potentially economic mineralisation is summarised in Table 13.3, below, which is used as the basis for the comminution circuit capacity evaluation for the Endako Restart Project. The Endako potentially economic mineralisation is considered soft with respect to resistance to impact breakage, as presented and compared against global benchmarks in Figure 13.2, below, and medium with respect to resistance to ball milling breakage. The Endako potentially economic mineralisation is considered medium abrasive.

TABLE 13.3
GRINDABILITY TEST SUMMARY FOR THE ENDAKO ORE

Sample Name	SPI		Work Indices (kWh/t)			AI
	min	kWh/t	AWI	RWI	BWI	g
Endako Ore	34.6	4.9	12.9	11.8	14.8	0.359

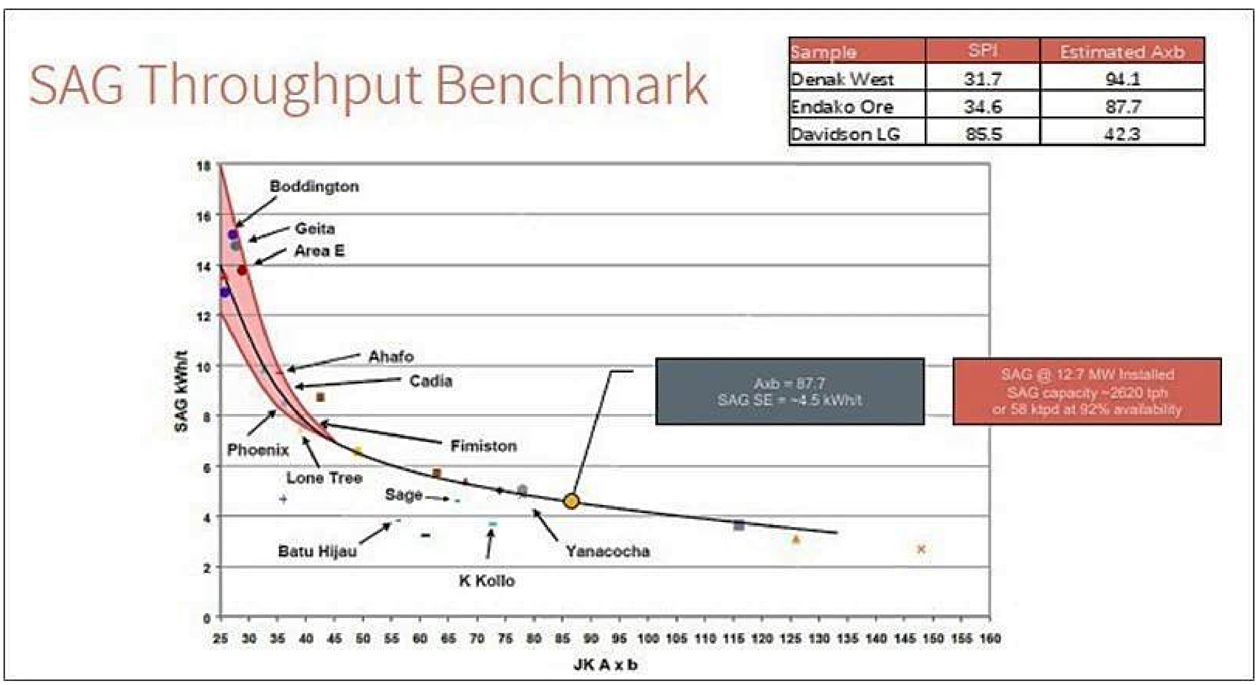


Figure 13.2 SAG Throughput Benchmark – Endako
***Source:** Amec Foster Wheeler*

The Endako in-pit crusher was modelled and simulated in the Metso-Outotec Bruno™ simulator. As shown in Figure 13.3, below, Hatch performed different simulations with three different ROM feed particle size scenarios, namely 1,000 mm medium, 800 mm medium and 800 mm fine. The simulation exercise suggests that the in-pit crusher at the Endako Mine has adequate capacity to achieve the plant nameplate (52 kilo tonnes per day), provided that the crushing circuit be maintained at 75% operating time or higher. The finer ROM feed particle sizes would improve the crushing operation to allow higher throughput capacity or finer product size.

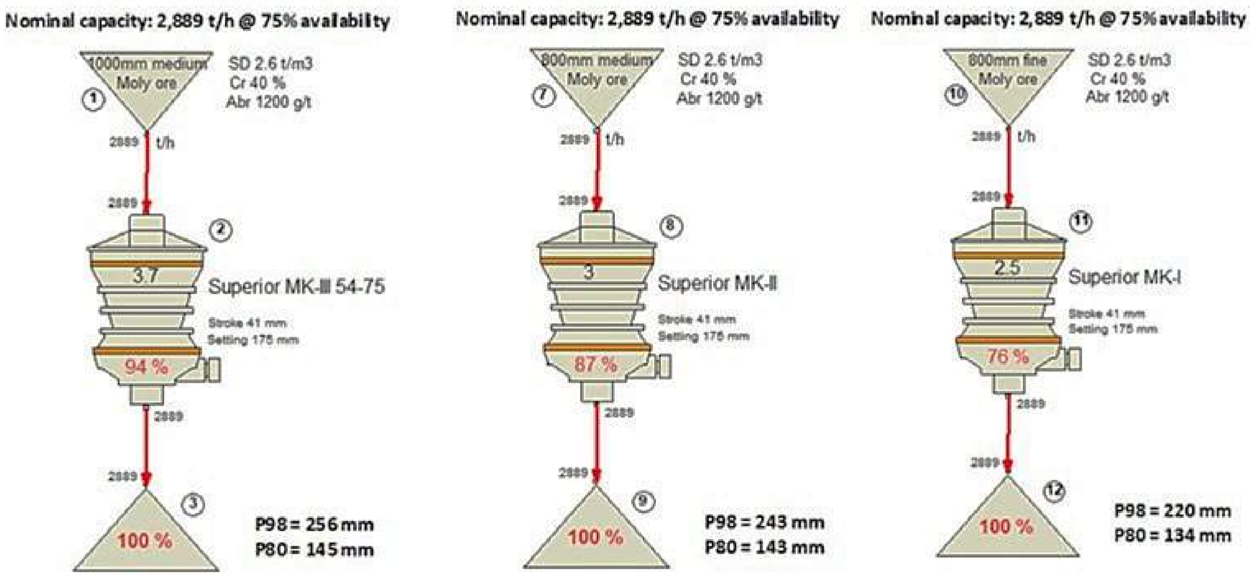


Figure 13.3 Bruno Simulation of the Endako In-pit Crusher
Source: *Hatch 2022*

Hatch utilises the Morrell's SMC™ power-based model (2004, 2009) for the prediction of the comminution circuit specific energy (kWh/t) based on the hardness characteristics of the potentially economic mineralisation. The SMC™ model represents a widely used and robustly developed method for comminution circuit design. The SMC™ model estimates the required total installed power for the comminution circuits based on the ore breakage characteristic, throughput and grind size target. Table 13.4, below, shows the SMC™ predicted throughput rate based on the test work completed for the 2014 Endako Expansion Project. The total comminution motor power requirement is predicted to be 24,753 kW for 52,000 tpd at 200 µm grind size. As can be seen, the existing Endako plant has a total installed motor power of 29,832 kW, including the SAG mill, pebble crusher and ball mills. This represents a 20% higher installed comminution power than that required by SMC™ calculation for the Endako ore.

TABLE 13.4 ENDAKO RESTART THROUGHPUT PREDICTION USING SMC™ MODEL		
Description	**Units**	**Endako Restart**
Production		
Annual Throughput	million t/a	18.98
Daily Throughput	t/d	52,000
Grinding Hourly Throughput at 92% Availability	t/h	2,355
Primary Crusher Product, 80% Passing	mm	150
Final Grind Size, 80% Passing	µm	200
Ore Characteristics		
Specific Gravity, SG	g/cm^3	2.60
JK Axb Parameter (Inferred)	-	87.7
Bond Ball Work Index (BWI)	kWh/t	14.9
Total Circuit Specific Energy		
Wa (Pinion)	kWh/t	5.4
Wb (Pinion)	kWh/t	3.9
Wc	kWh/t	0.15
Wt (Tumbling)	kWh/t	9.3
Wt (Total Pinion)	kWh/t	9.5
Total Power Requirement (Motor)	**kW**	**24,753**
Total Installed Power (SAG + Pebble + Ball Mill)	**kW**	**29,832**

Source: *Concentrator Support Ltd., 2025*

Table 13.5, below, summarises the statistics for the daily throughput and circuit utilisation of the Endako operation from January 2014 to July 2014. The average daily throughput was below the design target of 52,000 tonnes per day, largely due to circuit ramp-up, low plant availability and utilisation that were associated with initial operational issues. It was noted that the maximum recorded daily throughput was 88,094 tonnes per day, demonstrating the capacity potential of the milling circuit for processing the Endako potentially economic mineralisation.

TABLE 13.5 SUMMARY STATISTICS OF PRODUCTION IN 2014					
Description	**Units**	**Average**	**Min**	**Max**	**Std Dev**
Daily Tonnes Milled	dry t/d	44,164	-	88,094	21,314
Daily Utilisation	%	84	-	100	26

Source: *Hatch, 2022*

Figure 13.4, below, shows the distribution of the calculated hourly throughput from the 2014 Endako production data, with an average throughput of 2,288 tonnes per hour (t/h) (equivalent to daily throughput of 50,520 tonnes per day at 92% availability, which is around 2.8% lower than the design throughput of 2,355 t/h (or 52,000 tonnes per day at 92% availability). Figure 13.5, below, also shows that the Endako milling circuit had frequently exceeded the design hourly throughput. Figure 13.4, below, presents 2014 values for run-time versus actual.

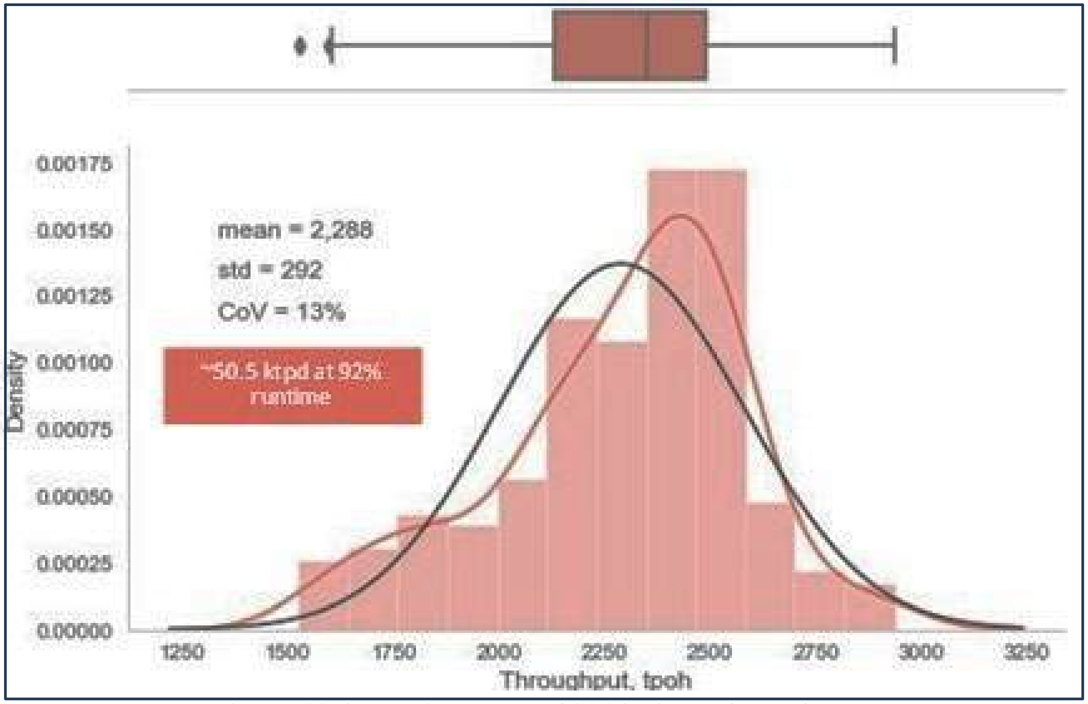


Figure 13.4 Histogram of 2014 Plant Throughput
***Source:** Hatch, 2022*

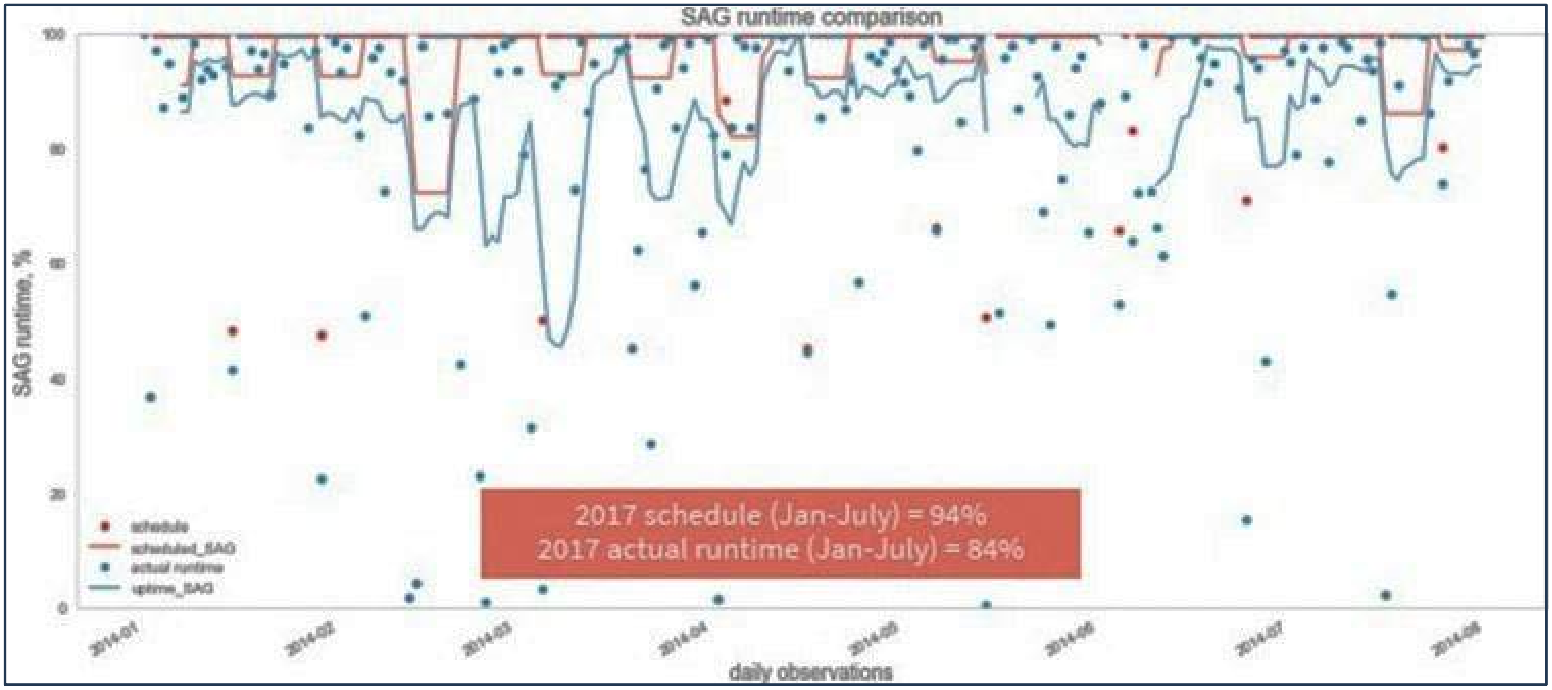


Figure 13.5 Mineral Processing and Metallurgical Testing
***Source:** Hatch, 2022*

Some identified issues from discussions with the site personnel are as follows. Addressing those issues would ensure the plant be maintained at the design production target or beyond.

- Raptor™ 1100 pebble crusher was rarely operating due to low pebble feed rate; thus, operation had been constantly recirculating the pebbles back to the SAG mill via the bypass chute and conveyor, resulting in excessive wear on the bypass chute. This consequently required weekly replacement to a patch on the bypass chute, as it was not designed to handle constant pebble recirculation. Possible mitigation solutions could consist of pebble crushing operation in batch mode or reconfiguring the SAG circuit to sustain the pebble production to enable consistent pebble crushing operation, which would result in an improved circuit availability.

- SAG discharge trommel screen panels were frequently worn out due to high flow and high impact on the panels, leading to SAG mill downtime for panel replacement. It was also observed that unwanted materials (SAG media and oversize particles) were bypassing to the ball mill circuit causing issues on the cyclone feed pumps. It is recommended to consider alternative trommel screen panels, *i.e.*, ceramic panel or solid impact panel to reduce operation interruption due to panel failure.

- In the later part of the 2014 operation, the SAG mill run time was impacted due to running worn out liners and having to replace bolts that were breaking due to impacts and no lift on the liners. A new SAG mill liner and robust relining program would improve the overall SAG mill availability.

13.5 FLOTATION

The predicted metallurgical recovery is based on the available production data during the year of 2014, following the commencement of the Endako Expansion Project. Low availability days were removed from the data set. The statistics of the flotation performance is summarised in Table 13.6, below. The MoS_2 head grade in 2014 data was averaged at 0.07%.

TABLE 13.6
SUMMARY STATISTICS OF FLOTATION PERFORMANCE IN 2014

Description	Units	Average	Min	Max	Std Dev
Ore Molybdenite Grade (MoS_2)	%	0.070	0.040	0.110	0.01
Molybdenite Recovery (MoS_2)	%	75.3	52.2	88.6	5.9

Source: *Hatch, 2022*

Figure 13.6, below, shows the distribution of the reported MoS_2 recovery from the 2014 Endako production data, with the average recovery being 75.3%. Note that data with lower than 1,500 t/h throughput were filtered before the analysis, as they are not representative of normal operating conditions.

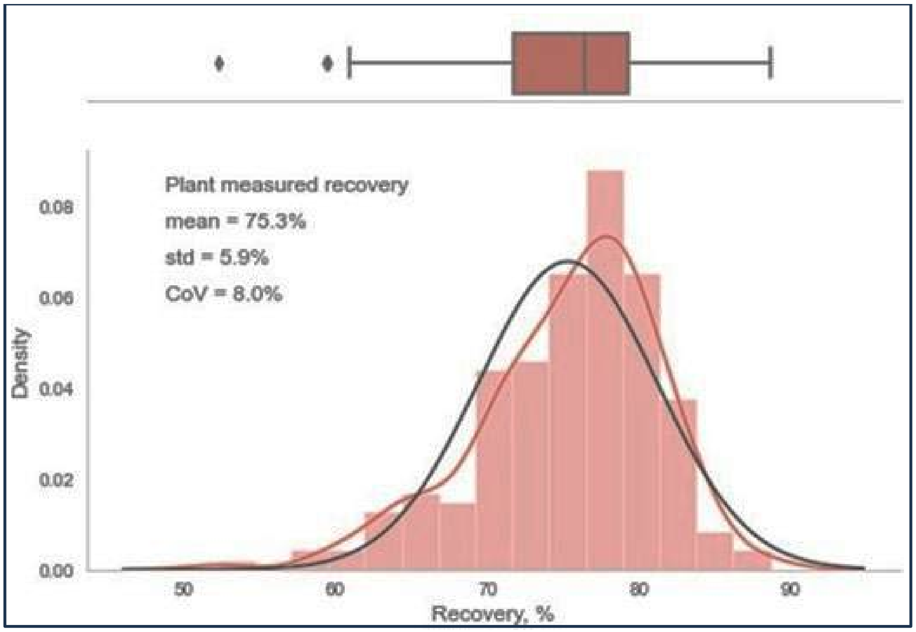


Figure 13.6 Histogram of 2014 Plant MoS_2 Recovery
***Source:** Hatch, 2022*

Through the review of historical process documents, it was found that the operation had developed an empirical model to forecast the MoS_2 recovery based on the MoS_2 head grade and ore source, as shown below.

1. Moly Recovery for Fresh Ore = -20.675 × MoS_2 %^2 + 5.0835 × MoS_2 % + 0.4852.

2. Moly Recovery for Stockpile Ore = -20.675 × MoS_2 %^2 + 5.0835 × MoS_2 % + 0.3852.

The model suggests that the flotation recovery of the stockpiled potentially economic mineralisation is expected to be 10% lower than the fresh ore due to oxidisation. The recovery relation is applicable up to 0.12% MoS_2 in the feed, for 80% recovery. For feed grades exceeding 0.12% MoS_2, the recovery should be capped at the 0.12% MoS_2 computed value.

Figure 13.7 and Figure 13.8, below show the difference between the actual reported recovery in 2014 and the forecasted recovery for the fresh ores only. This recovery model is the only model that Hatch had identified, and it is considered appropriate for the MoS_2 recovery projection for the Endako Restart Project.

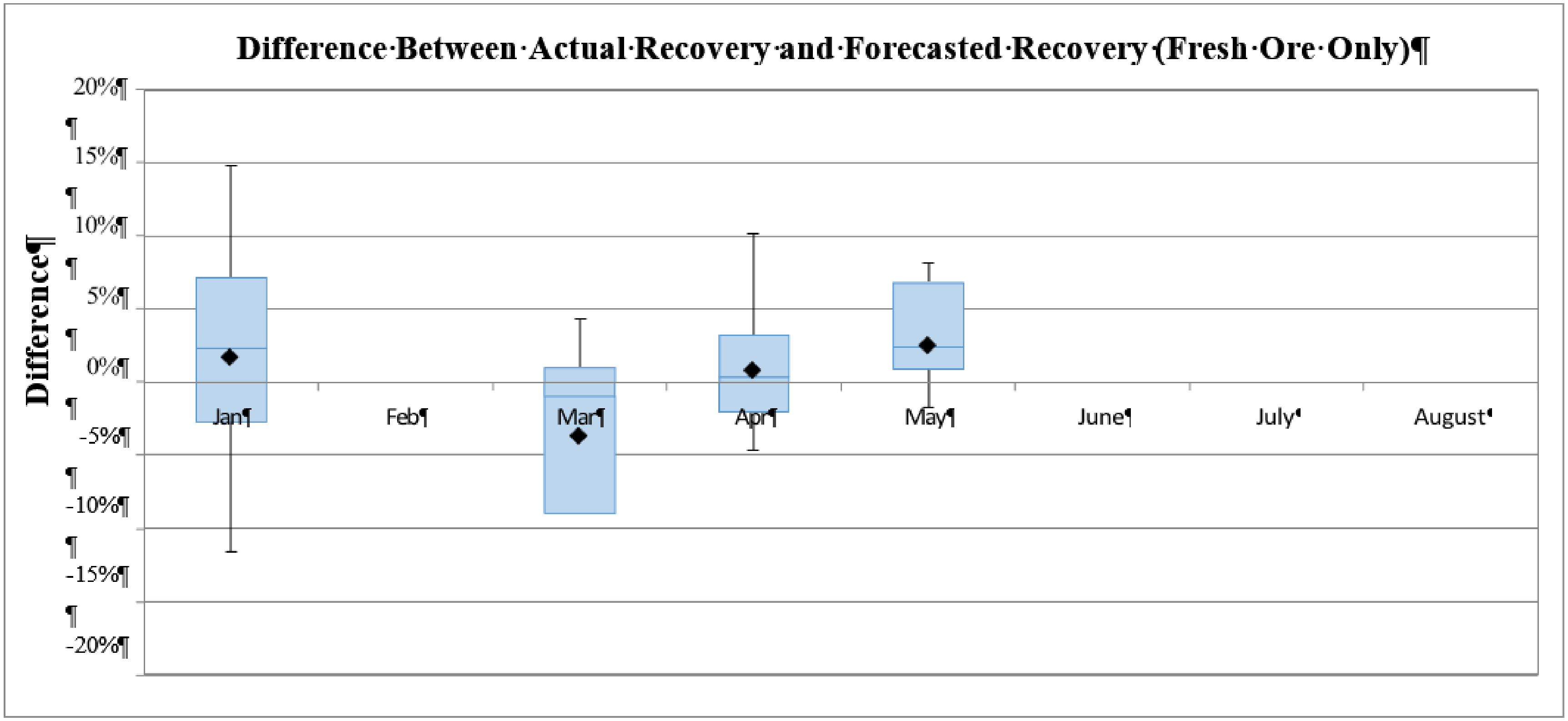
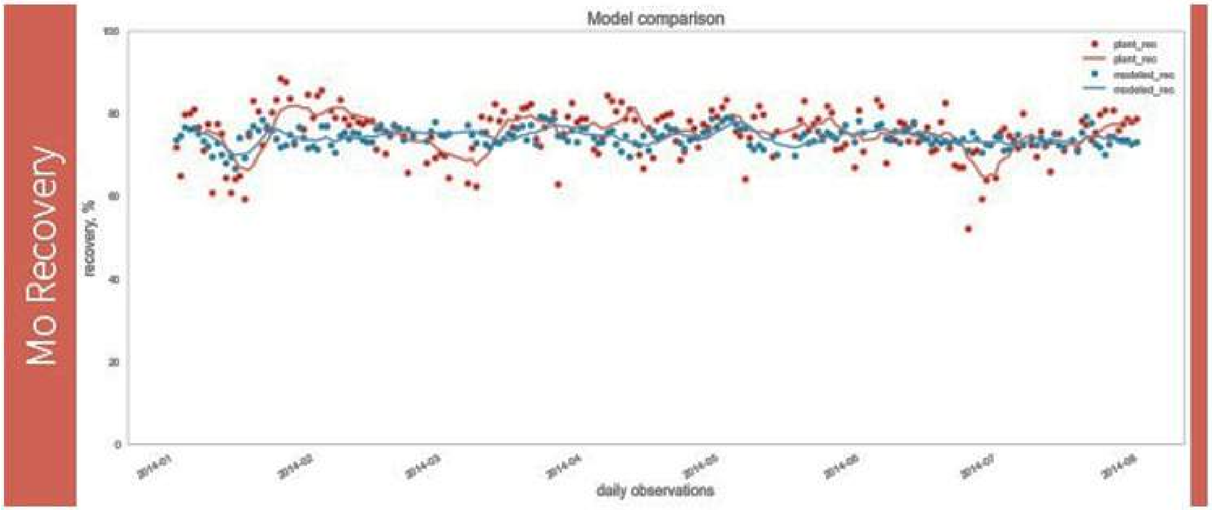


Figure 13.7 Model Comparison Against Actual

***Source:** Hatch, 2022*



	plant_rec	modeld_rec
count	182	182
mean	75	74.27
std	6.1	2.26
min	52.23	66.64
25%	71.7	72.91
50%	75.96	74.02
75%	79.07	75.68
max	88.57	79.24
cov	8.1%	3%

Figure 13.8 Model Comparison against Actual
***Source:** Hatch, 2022*

13.6 CONCENTRATE QUALITY

The historical MoS_2 concentrate grade is summarised in Table 13.7, below, which was in the range of 90% to 93.3% MoS_2 (from 25th percentile to 75th percentile).

TABLE 13.7
CONCENTRATE PRODUCTION STATISTICS IN 2014

Description	Units	Average	Min	Max	Std Dev
Molybdenite Concentrate Grade	%	91.5	82.7	95.8	2.1

Source: *Hatch, 2022*

13.7 CONCENTRATE DEWATERING

Molybdenum concentrate was historically dewatered successfully at Endako using a disc filter and dryer.

No test work has been performed or provided by Centerra for concentrate dewatering, either historical or new. Sizing of concentrate dewatering systems for the study is based on conservative values and off industry benchmarks.

It is recommended that additional equipment specific test work be completed on concentrate dewatering, *i.e.*, filtration and drying, in the next phase of work to confirm design criteria for final equipment selection.

14.0 MINERAL RESOURCE ESTIMATES

The resource model described in this section was completed by AMPL during the period between January 2025 and August 2025. The model is considered current as of September 1, 2025.

Mr. Finley Bakker, P.Geo. is the Qualified Person responsible for the Resource estimate. Mr. Bakker is a Qualified Person by virtue of education, experience and membership in a professional association. He is independent of the Company applying all the tests in Section 1.5 of the NI 43-101.

There appears to be no issues or factors that could materially affect the Mineral Resource estimate. This includes no issue involved with environmental permitting, legal, title, taxation, socio-economic, marketing, political, mining, metallurgical or infrastructure.

The mine is located in central British Columbia, Canada approximately 22 km west of the municipality of Fraser Lake. HxGN Mine Plan™ 3D (previously known as MineSight/Hexagon/MedSystem™) software was used for the Resource estimate. The metal of interest at the Endako Mine is molybdenum.

14.1 DATA

During September to November 2024, Endako staff provided AMPL with a data set consisting of:

- Drill data for the Endako Mine comprising the following:
  - Collar data;
  - Downhole surveys; and
  - Sample numbers and MoS_2 assays.

- Production blast hole data was provided in two files because a number of blast holes had more than one assay associated with the hole ID. The two files were:
  - Collar data; and
  - Sample data.

- Fifteen 3D domain wireframes for the Endako pit area; and

- Topography files consisting of:
  - The December 2014 rock surface topography;
  - The December 2014 scree;
  - A LiDAR™ topography dated August 2013; and
  - Topography pre-mining.

This was supplemented with information received from Mr. Desautels in March and April 2025. This included the work Mr. Desautels prepared for the Centerra Gold Internal Feasibility Study of 2018. Additional information was also gleaned from Assessment reports filed with the British Columbia mineral titles online. The entire Sojitz diamond drill hole database and blast hole data was obtained at a later date and compared to previous information and subsequently incorporated into the Resource model.

This data was distributed with various reports, scanned maps and strip logs. During the site visit, this data set was enhanced by the addition of the December year end pit surfaces covering the period between 2009 through to 2014, QA/QC data, Acid Base Accounting (ABA) data and the entire Sojitz database.

The above data set was imported into HxGN Mine Plan™ 3D, and all data was checked for overlapping, missing and negative length intervals. No erroneous data was detected affecting the primary database table used in the Resource estimation. The QP is satisfied that the data was acceptable for use in a Mineral Resource calculation.

The official data cut-off date for this resource estimate is **31 of December 2014** when the Endako Mine shut down, although the Resource was calculated in 2025 A total of 1,335 diamond drill holes (generally NQ in size) exist in the database and include 58,515 assays. The percussion drill hole database included 330 drill holes and 7,792 assays.

The drill data was supplemented with 31,784 recent blastholes and 259,071 archived blast holes. The database includes over 13 million feet of drilling or 2,500 miles or 4,000 km of drilling (see Table 14.1, below).

TABLE 14.1
DRILL HOLE SUMMARY HOLES USED IN RESOURCE ESTIMATION

Data Source	Number of Holes	Total Length (ft)	Number of Assays
Diamond Drill Holes	1,335	669,175	58,515
Percussion Holes	330	88,700	7,797
Blast Holes	31,784	1,298,770	31,972
Archived Blast Holes	259,071	11,237,309	259,071
Total	**292,520**	**13,293,954**	**357,355**

Source: *AMPL, 2025*

14.1.1 Sojitz versus Endako Data

The Sojitz database was compared with the database provided by Centerra. Overall, they compared favourably but it became apparent that there was an error in the database provided by Centerra. A number of assays were truncated (not rounded off). For example, both 0.027 and 0.022 were truncated to 0.02. This did not occur in any systematic or consistent manner. These numbers were compared to hard copies and it was evident that an error had occurred somewhere in the importing of the data. The results are summarised in the Table 14.2, below. In all cases, the grade was underreported, which would be the case if assays are truncated rather than rounded. The Sojitz data was subsequently used in the building of the model. It should be noted that the majority of the truncating occurred at lower values and to the third decimal place and as such, while potentially under reporting the Mineral Resource, the overall impact would only be seen at lower cut-offs.

TABLE 14.2
DRILL HOLE COMPARISON OF SOJITZ VERSUS CENTERRA GOLD DATA

Hole Series	Number of Assays	% Difference in Grade
06 Series Holes	3,239	9.34
07 Series Holes	3,179	8.92
08 Series Holes	2,547	8.15
10 Series Holes	4,358	6.64
11 Series Holes	7,409	8.42
12 Series Holes	947	10.15
67 Series Holes	484	0.04
BG Holes	157	0.11
S Series Holes	22,875	0.14
S2 Series Holes	13,665	0.00
S-01-04 Holes	682	10.75
Total Assays	**59,542**	**3.21**

Note: This differs slightly from reported assays used; several holes were repeated in this analysis.

Source: *AMPL, 2025*

14.2 GEOLOGICAL INTERPRETATION

The Endako deposit consists of an elongated stockwork of quartz-molybdenite veins within the Endako quartz monzonite. Individual veins have limited projection but the vein systems and associated stockwork can be traced throughout the deposit. Basalt faults are major zones of post-mineralisation dislocation that divide the deposits into five structurally distinct zones. All major structures intersect the north-dipping South Boundary Fault, which is considered to be the basic controlling structure of the ore deposit. The basic geological zones are shown in Figure 14.1, below.

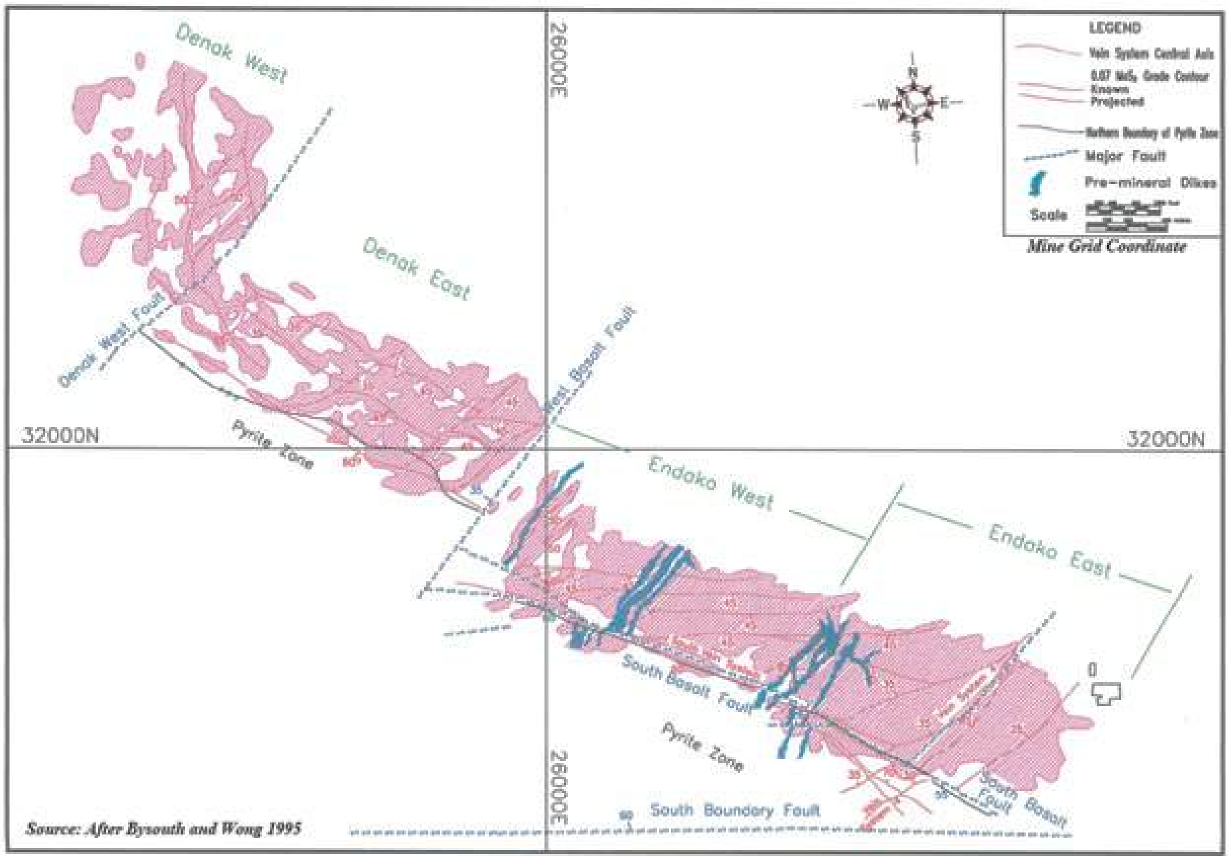


Figure 14.1 Geological Boundaries
***Source:** Bysouth and Wong, 1995*

This basic model was then supplemented by wireframes defined by Endako Mine Staff and modified by AMPL (2025) (Figure 14.2 to Figure 14.8, below).

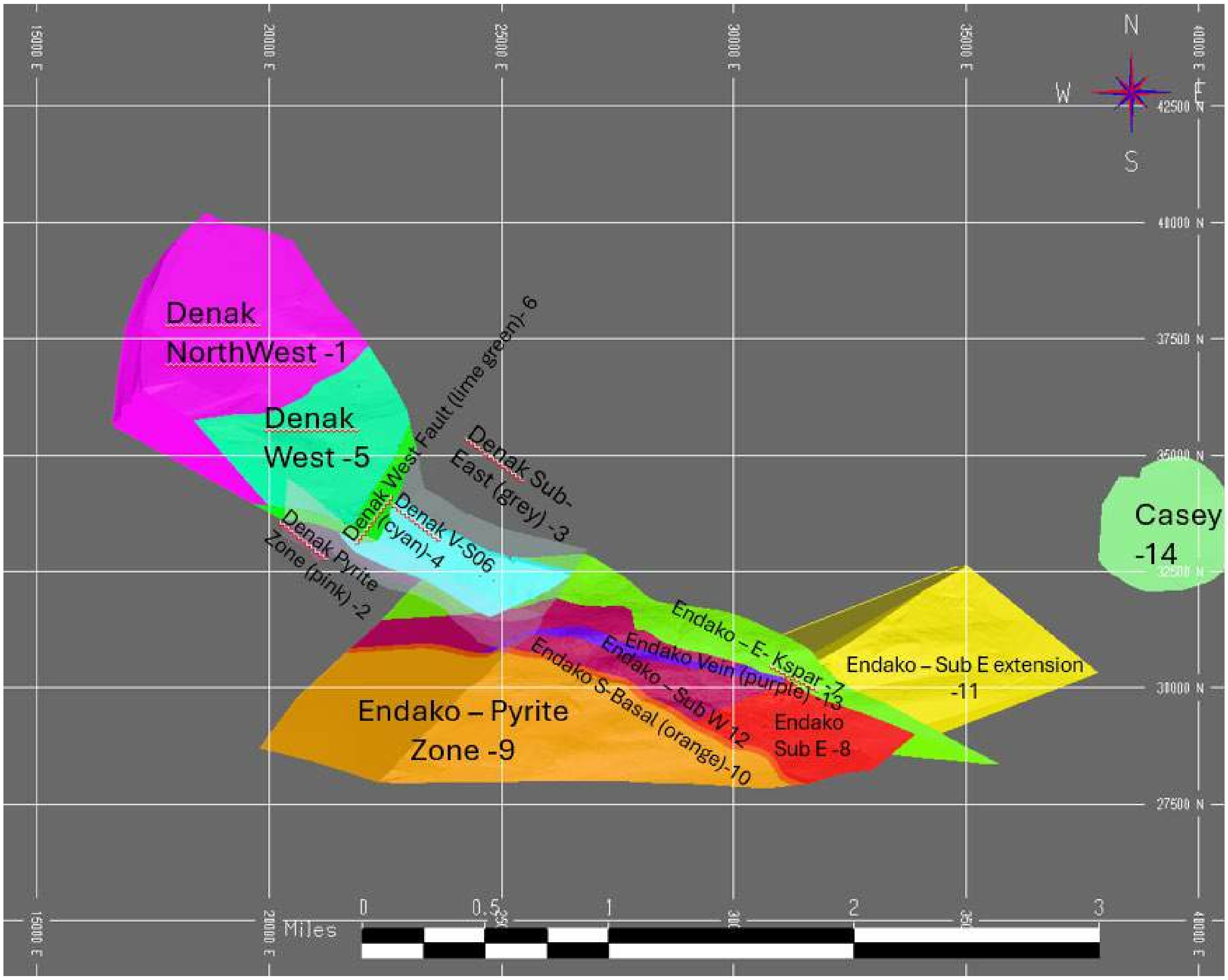


Figure 14.2 Geological and Resource Boundaries

Source: *AMPL, 2025*

Endako East (Endako – K-spar, Endako – Sub, Endako Sub E Kspar, Endako Vein): Mineralisation is hosted in northeasterly striking vein systems that dip at low angle to the northwest.

***Endako West* (Endako sub-W, Endako, Endako S-Basal, Endako Pyrite):** Vein systems in this domain typically strike at 110 degrees and dip 60 degrees to the southwest.

Figure 14.3 Endako Pyrite Zone Hole 06-07
Source: *Taiga Consultants, 2006*

Figure 14.4 Endako Pyrite Zone Hole 06-06 at 501.2 ft
Source: *Taiga Consultants, 2006*

Figure 14.5 Endako S Basal Zone in Hole S06-07
***Source:** Taiga Consultants, 2006*

Figure 14.6 Endako S Basal Zone in Hole S06-06
***Source:** Taiga Consultants, 2006*

Figure 14.7 Endako Extension Zone Hole S06-06
Source: *Taiga Consultants, 2006*

Figure 14.8 Endako Extension Zone Hole S06-08
Source: *Taiga Consultants, 2006*

***Denak East* (Denak V-S06, Denak Sub-East, Denak Pyrite):** The mineralisation at Denak East is trending between 110 and 130 degrees with a southwesterly dip. The development of vein systems is weaker throughout the Denak Zone.
Denak West: The mineralisation at Denak West is trending between 170 degrees with a southwesterly dip. The development of vein systems is weaker throughout the Denak Zone.
Denak Northwest: Appears to be an extension of the Denak West domain. Mineralisation strikes approximately due north and may be separated from Denak West by a fault.
Casey: The mineralisation appears to be strongest along an east-northeast trend, possibly representing the intersection of two or more major vein systems.

The domain and code used in the report is summarized in Table 14.3, below.

TABLE 14.3
DOMAIN NAME AND DOMAIN CODE

Legend		
Denak Northwest	=	1
Denak Pyrite Zone	=	2
Denak Sub-East	=	3
Denak v-S06	=	4
Denak West	=	5
Denak West Fault	=	6
Endako-E K-spar	=	7
Endako Sub E	=	8
Endako Pyrite	=	9
Endako S-Basal	=	10
Endako Sub-E Extension	=	11
Endako Sub W	=	12
Endako Vein	=	13
Casey	=	14

Source: *AMPL, 2025*

14.3 TOPOGRAPHY

Topography was provided by Endako Mine as a 3D surface and contour lines. The latest LiDar™ surface provided was dated August 2013. This surface is complete and displays an accurate representation of the dumps and tailings; however, the bottom of the Endako and Denak Pits, shown on the surface, is representative of the water level at the time the survey was conducted. As such, it should not be relied on to provide an accurate representation of the current pit volumes. This information was supplemented with blasted information.

There was an original pre-mining topography provided, which was assembled from historic government contour maps when no surface disturbance existed. This surface was manipulated to increase the coverage to the extent of the Resource model. The data is not nearly as accurate or precise as the Lidar™ information and could only be used as general outlines.

A review of the literature indicates that the December 2014 scree surface represents the open pit and dumps merged with the LiDAR™ surface and excludes the water in the pit. There is, at times, considerable difference between the December 2014 rock bottom surface, which represents the lowest elevation mined stripped of dumps, tailings and water. These surfaces are usually provided for the area surrounding the open pit only and are derived from 3D wall scans of the pit during mining.

The difference is attributed to the difference between what was blasted and mined and the final repose of the pit. As such, the rock bottom was used as the final solid in the pit and the difference is sloughed material of unknown grade (see Figure 14.9 and Figure 14.10, below).

Figure 14.9 View of Pit Wall Showing Difference Between Skree and Rock Profiles
***Source:** AMPL, 2025*

Figure 14.10 View of Pit Wall Showing Difference Between Skree and Rock Profiles
***Source:** AMPL, 2025*

14.4 OVERBURDEN

No overburden surface was provided by Endako Mine, which is not a concern for the areas that have been extensively mined. The Denak Northwest area and the Casey Zone were never stripped. For those two zones, an overburden/till surface was created in HxGN Mine Plan™ 3D. This surface was used to limit the inclusion of overburden. For the other lenses it was assumed that topography was also bedrock.

The surface was based on the depth of the first assay for holes that were drilled from the original pre-mining topography. This method assumes that the first assay was collected at the bottom of the casing, and it also assumes that the casing was carried through the till and overburden layers only. In some holes, the casing may be deeper due to fault gouge. A few drill holes were omitted since the depth of the first assay did not seem to correlate with the nearby holes. It is noted that the lithology from the drill logs should normally be available in the mine database but Endako Mine did not digitize this information when they converted the information from the paper logs.

14.5 DIAMOND DRILL HOLE DATA

14.5.1 Diamond Drill Downhole Collars

Diamond drill hole assays were received as ".csv" files, which were provided by Endako personnel. They were received as collar, survey and assay files. Examples of each are given in Table 14.4, below.

TABLE 14.4
EXAMPLE OF A COLLAR FILE

HOLE-ID	LOCATION	LOCATION	LOCATION	LENGTH	JMAREK	RESOURCE	PROGRAM
06-001	26026	29844	3103	717	yes	yes	TC-2006
06-002	25850	29999	3108	727	yes	yes	TC-2006
06-003	27135	29062	3286	747	yes	yes	TC-2006
06-004	25850	29999	3108	447	yes	yes	TC-2006
06-005	28887	28487	3092	707	yes	yes	TC-2006
06-006	28324	28826	3051	806	yes	yes	TC-2006
06-007	28133.4	28916.8	3056.1	917	yes	yes	TC-2006
06-008	26817.7	29686.5	2963.9	768	yes	yes	TC-2006
06-009	27549	29103.6	3071.6	896	yes	yes	TC-2006
06-010	20910.73	36581.49	2989.57	540	yes	yes	TC-2006
06-011	20498.17	36542.95	2983.01	507	yes	yes	TC-2006
06-012	20674.93	36561.34	3009.25	607	yes	yes	TC-2006
06-013	20692.63	36407.55	3002.69	507	yes	yes	TC-2006
06-014	20515.59	36399	2996.13	507	yes	yes	TC-2006
06-015	20484.12	36132.12	3015.81	507	yes	yes	TC-2006
06-016	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

Source: *AMPL, 2025*

14.5.2 Diamond Drill Downhole Surveys

Table 14.5, below, shows an example of the survey file source used in the report.

TABLE 14.5
EXAMPLE OF SURVEY FILE

HOLE-ID	DISTANCE	AZIMUTH	DIP
06-001	0	7	-60
06-002	0	7	-60
06-003	0	7	-60
06-004	0	323	-60
06-005	0	7	-60
06-006	0	7	-60
06-007	0	7	-60
06-008	0	7	-75
06-009	0	7	-75
06-010	0	0	-60
06-011	0	0	-60
06-012	0	0	-60
06-013	0	0	-60
06-014	0	0	-60
06-015	0	0	-75
06-016	0	0	-60
06-017	0	0	-60
06-018	0	0	-90
06-019	0	0	-90

Source: *AMPL, 2025*

Note that most of the holes did not have downhole surveys and the ones that did typically used acid tests. The QP feels that on the scale of the deposit this has no significant impact.

Readers may be concerned that there was no way to physically inspect the downhole surveys, but as in the case of most properties, it is impossible to go back down existing drill holes without remobilizing a diamond drill. In some cases, physical artifacts are available to be examined and compared to the digital data (*e.g.*, photos taken by SperrySun), in others, the data is simply transcribed onto the drill log. It must be emphasised that there is generally no way to absolutely know the precise location of a diamond drill hole. Again, readers may be concerned that most drill holes only used acid tests or no tests at all. The reader should be aware that these acid tests are still accepted as a valid method of determining the orientation of a hole, with health concerns using hydrofluoric acid notwithstanding.

Readers should be aware that acid tests only record dip and not azimuth changes in a diamond drill hole. The majority of the diamond drilling and all the blast holes are vertical These holes tend to "pig tail" or corkscrew rather than deviate to the up and right.

A number of diamond drill holes that were not drilled vertically did have downhole surveys taken. An example is given below in Figure 14.11, below. The holes did not deviate to any extent.

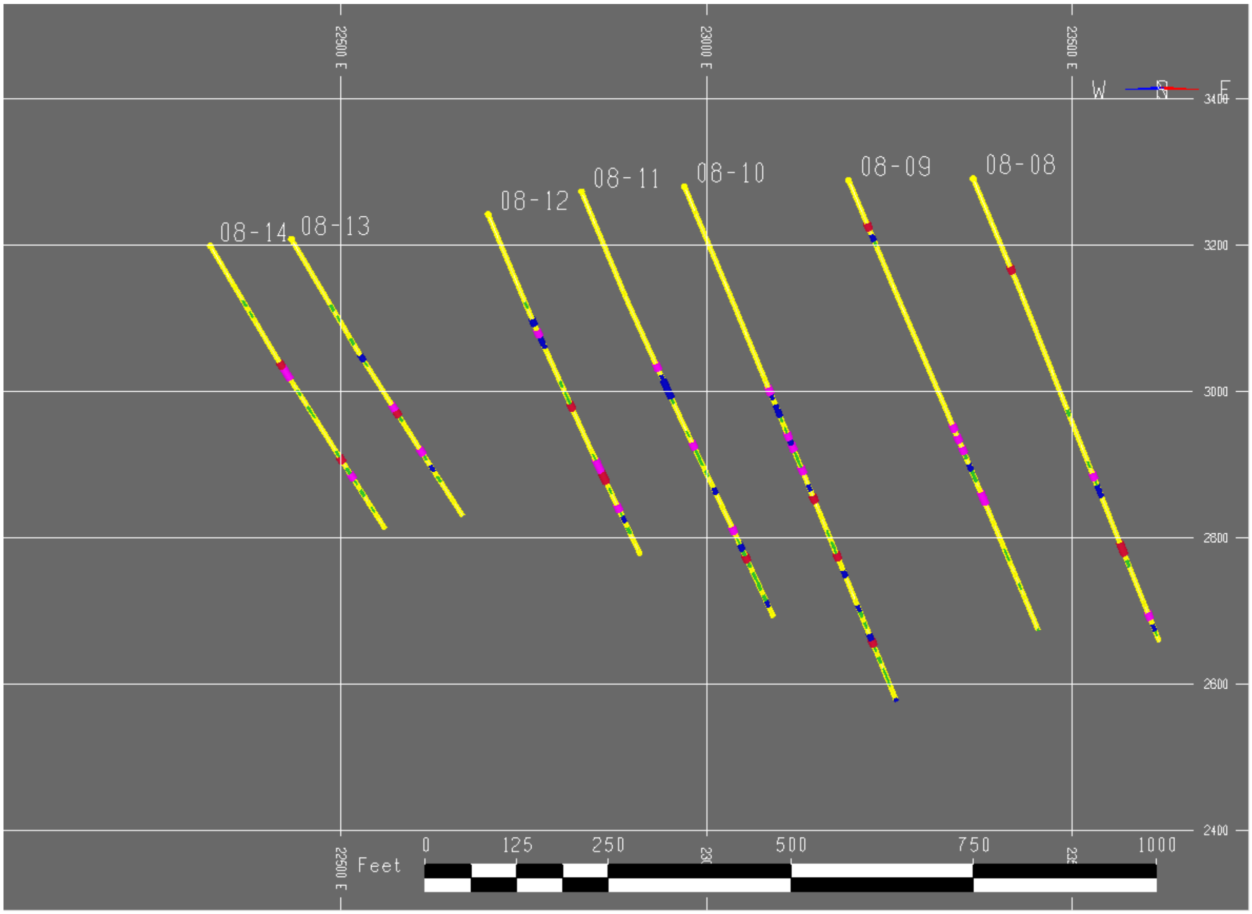


Figure 14.11 Sectional Plot Looking North of Diamond Drill Holes with Downhole Surveys
Source: *AMPL, 2025*

As such, it is the opinion of the QP, the holes that did not have downhole surveys or only acid tests will have negligible impact on the resource and shape of the deposit and in all likelihood have no impact based on the distance between drill holes and the size of the deposit.

14.5.3 Diamond Drill Downhole Assays

Table 14.6 , below, shows examples of the survey file used in this report.

TABLE 14.6
EXAMPLE OF SURVEY FILE

HOLE-ID	FROM	TO	MOS2	SMPID
06-001	12	20	0.009	13201
06-001	20	30	0.02	13202
06-001	30	40	0.06	13203
06-001	40	50	0.009	13204
06-001	50	60	0.01	13205
06-001	60	70	0.01	13206
06-001	70	80	0.01	13207
06-001	80	90	0.04	13208
06-001	90	100	0.02	13209
06-001	100	110	0.02	13210
06-001	110	120	0.02	13211
06-001	120	130	0.01	13212
06-001	130	140	0.005	13213
06-001	140	150	0.01	12601
06-001	150	160	0.009	13215
06-001	160	170	0.006	13216
06-001	170	180	0.01	13217
06-001	180	190	0.03	13218
06-001	190	200	0.009	13219
06-001	200	210	0.02	13220
06-001	210	220	0.007	13221
06-001	220	230	0.01	13222
06-001	230	240	0.02	13223
06-001	240	250	0.03	13224
06-001	250	260	0.01	13225
06-001	260	270	0.007	13226

Source: *AMPL, 2025*

These files were then combined to create a single file for input into HxGN Mine Plan™ 3D using the program Consca.dat (see Figure 14.12 to Figure 14.14, below).

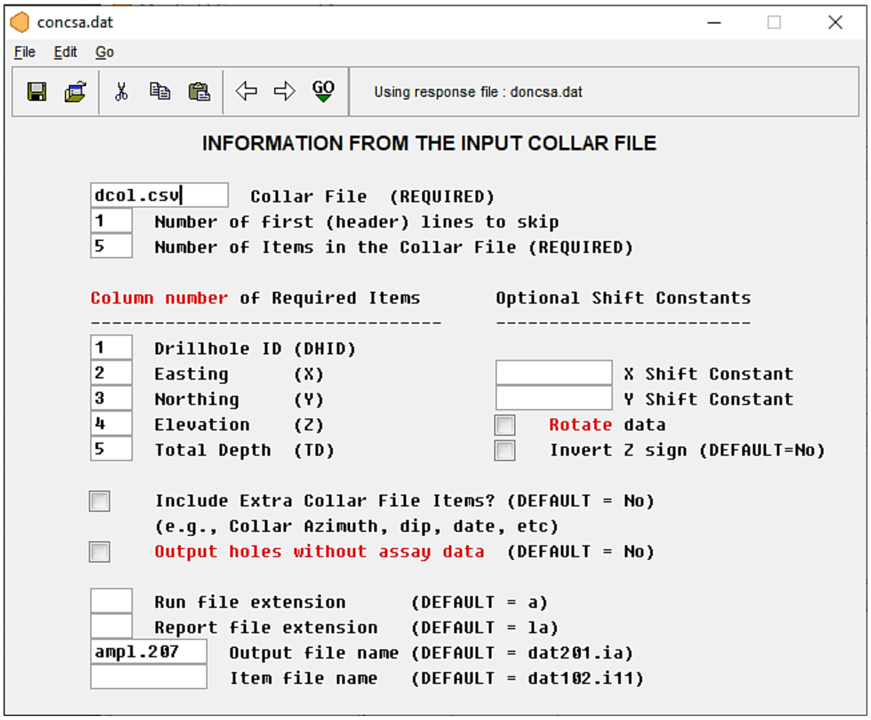


Figure 14.12 HxGN Mine Plan™ 3D Input File for Diamond Drill Holes
***Source:** AMPL, 2025*

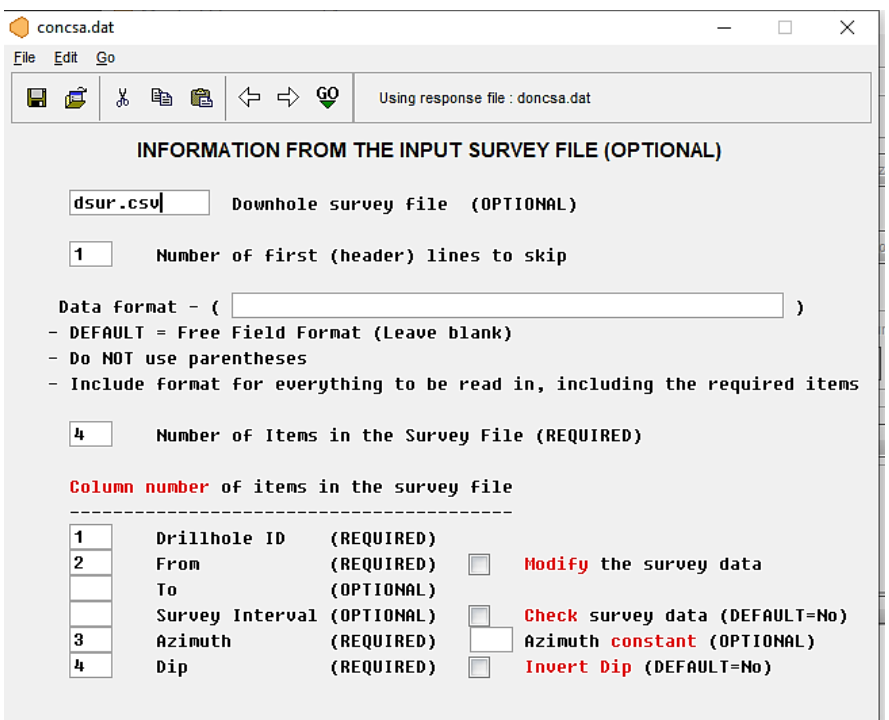


Figure 14.13 HxGN Mine Plan™ 3D Input File for Diamond Drill Holes
***Source:** AMPL, 2025*

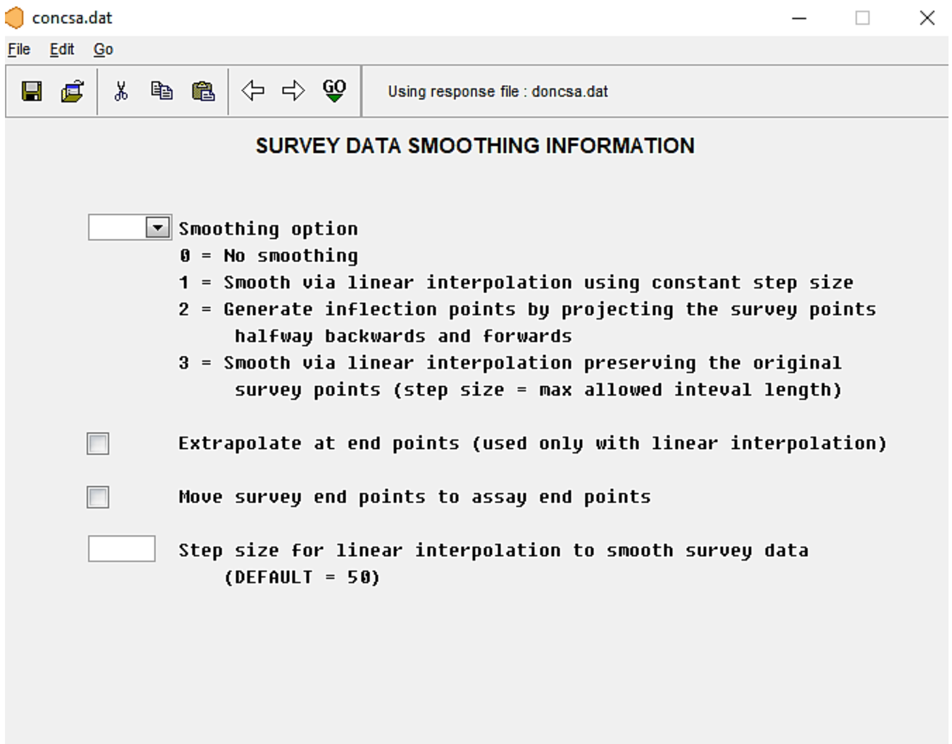


Figure 14.14 HxGN Mine Plan™3D Input File for Diamond Drill Holes
Source: *AMPL, 2025*

Care must be taken when using this panel – depending on which option is selected, drill hole toes can move significantly. For the case of this option, 0 was used because it tends to mimic traditional hand drawn drill hole traces.

This creates an input file of which an example is given below (Table 14.7, below).

TABLE 14.7
EXAMPLE OF DIAMOND DRILL HOLE INPUT FILE INTO HXGN MINE PLAN™ 3D

06-001	26026.00	29844.00	3103.00	7.00	-60.00	717.00
06-001						
06-001	0.00	12.00	12.00	-1.0000	-1.0000	-1
06-001	12.00	20.00	8.00	0.0090	13201.0000	-1
06-001	20.00	30.00	10.00	0.0200	13202.0000	-1
06-001	30.00	40.00	10.00	0.0600	13203.0000	-1
06-001	40.00	50.00	10.00	0.0090	13204.0000	-1
06-001	50.00	60.00	10.00	0.0100	13205.0000	-1
06-001	60.00	70.00	10.00	0.0100	13206.0000	-1
06-001	70.00	80.00	10.00	0.0100	13207.0000	-1
06-001	80.00	90.00	10.00	0.0400	13208.0000	-1
06-001	90.00	100.00	10.00	0.0200	13209.0000	-1
06-001	100.00	110.00	10.00	0.0200	13210.0000	-1
06-001	110.00	120.00	10.00	0.0200	13211.0000	-1
06-001	120.00	130.00	10.00	0.0100	13212.0000	-1
06-001	130.00	140.00	10.00	0.0050	13213.0000	-1
06-001	140.00	150.00	10.00	0.0100	12601.0000	-1
06-001	150.00	160.00	10.00	0.0090	13215.0000	-1
06-001	160.00	170.00	10.00	0.0060	13216.0000	-1
06-001	170.00	180.00	10.00	0.0100	13217.0000	-1
06-001	180.00	190.00	10.00	0.0300	13218.0000	-1
06-001	190.00	200.00	10.00	0.0090	13219.0000	-1
06-001	200.00	210.00	10.00	0.0200	13220.0000	-1
06-001	210.00	220.00	10.00	0.0070	13221.0000	-1
06-001	220.00	230.00	10.00	0.0100	13222.0000	-1
06-001	230.00	240.00	10.00	0.0200	13223.0000	-1
06-001	240.00	250.00	10.00	0.0300	13224.0000	-1
06-001	250.00	260.00	10.00	0.0100	13225.0000	-1
06-001	260.00	270.00	10.00	0.0070	13226.0000	-1
06-001	270.00	280.00	10.00	0.0070	13227.0000	-1
06-001	280.00	290.00	10.00	0.0300	13228.0000	-1
06-001	290.00	300.00	10.00	0.0100	13229.0000	-1
06-001	300.00	310.00	10.00	0.0700	13230.0000	-1
06-001	310.00	320.00	10.00	0.0800	13231.0000	-1
06-001	320.00	330.00	10.00	0.0200	13232.0000	-1
06-001	330.00	340.00	10.00	0.0800	13233.0000	-1

Source: *AMPL, 2025*

Dat.207 used to input drill hole data into HxGN Mine Plan™ 3D using procedure M201 (Figure 14.15, below).

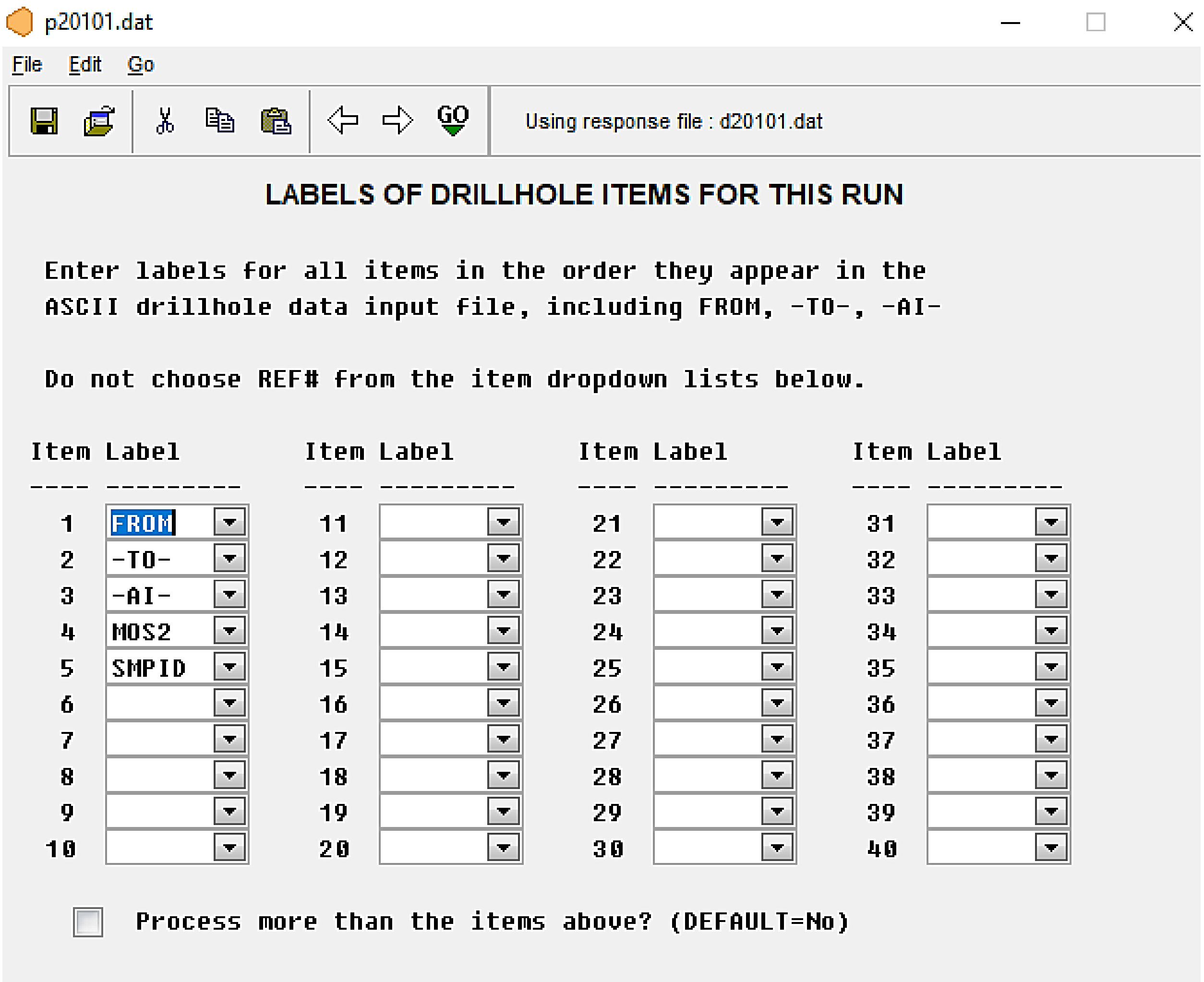


Figure 14.15 ***One of the Input Panels Used in M201 to Input Data into HxGN Mine Plan™ 3D***
Source: *AMPL, 2025*

The 3D views of diamond drill holes can then be created in HxGN Mine Plan™ 3D (Figure 14.16, below).

Figure 14.16 ***Example of Creating Diamond Drill Hole Views***
Source: *AMPL, 2025*

It will be necessary to refer to the HxGN Mine Plan™ 3D Program to determine how and where this is to be done. Zeros were substituted for any missing assays (-1 value).

The diamond drill holes were then flagged with one of the 14 lithological units indicated in Figure 14.17 to Figure 14.19, below. Non-assigned areas were given the code of 15.

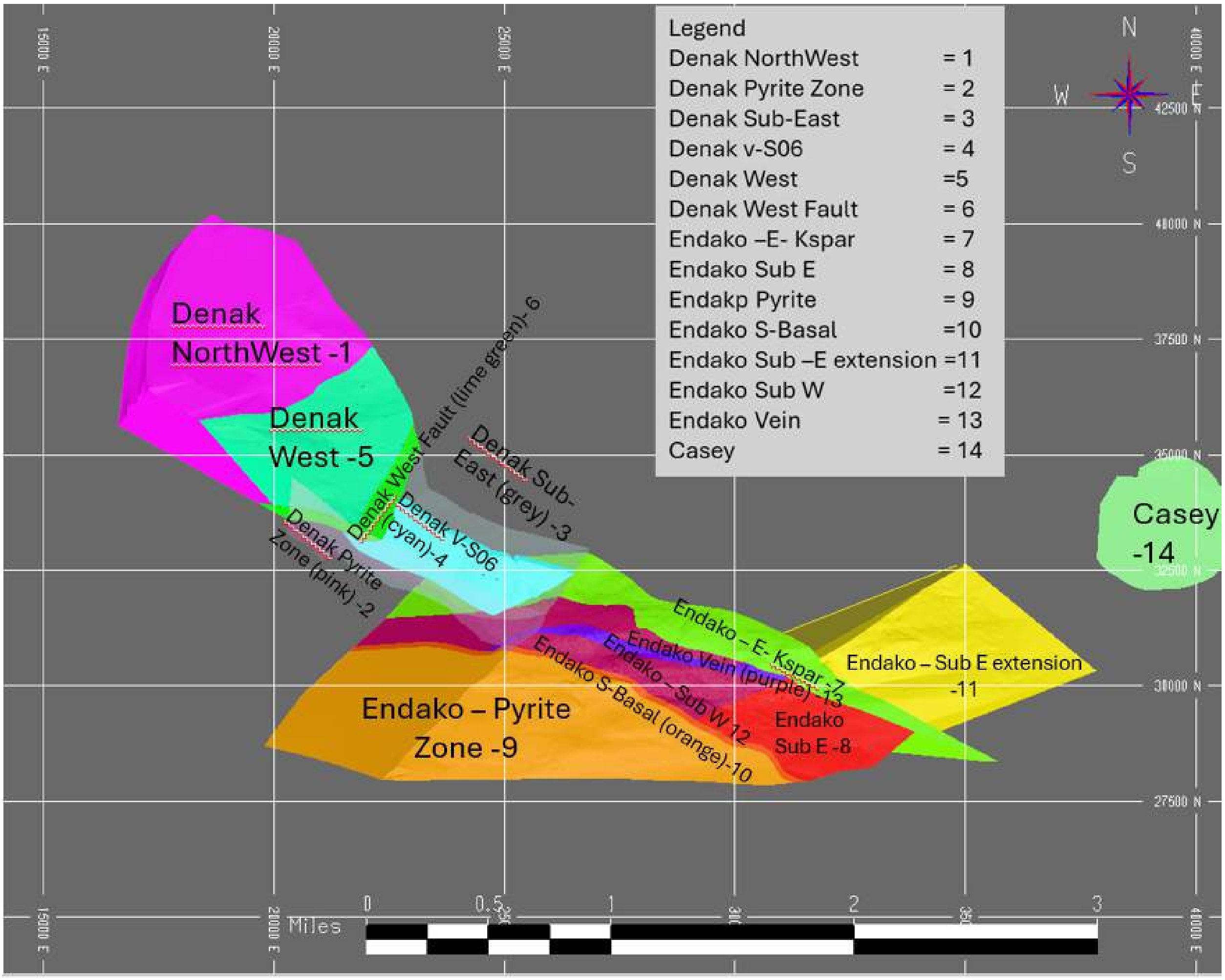


Figure 14.17 Lens/Wireframe Codes Assigned to Diamond Drill Holes (and Blast Holes) HxGN Mine Plan™ 3D

Source: *AMPL, 2025*

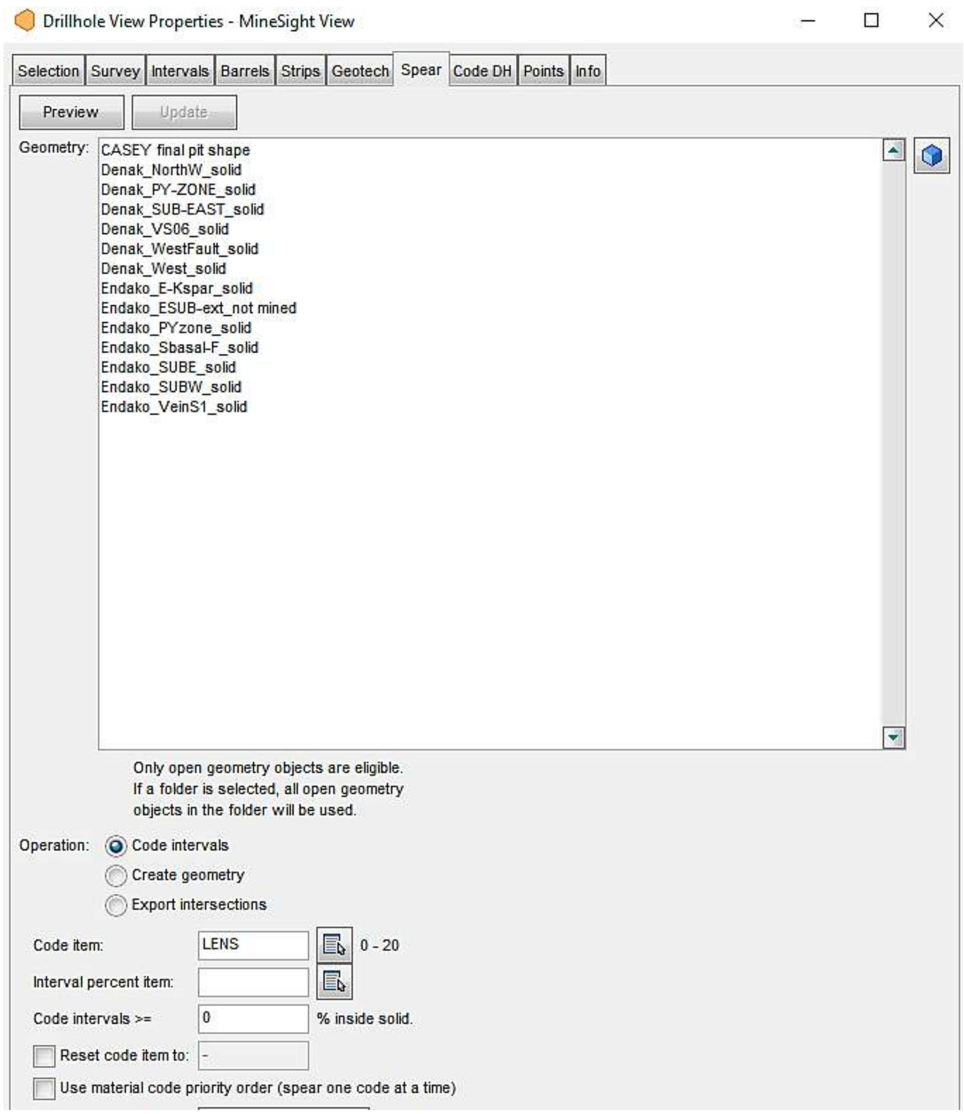


Figure 14.18 HxGN Mine Plan™ 3D Panel Showing which Lithological Units Were Used for "Spearing"

Source: *AMPL, 2025*

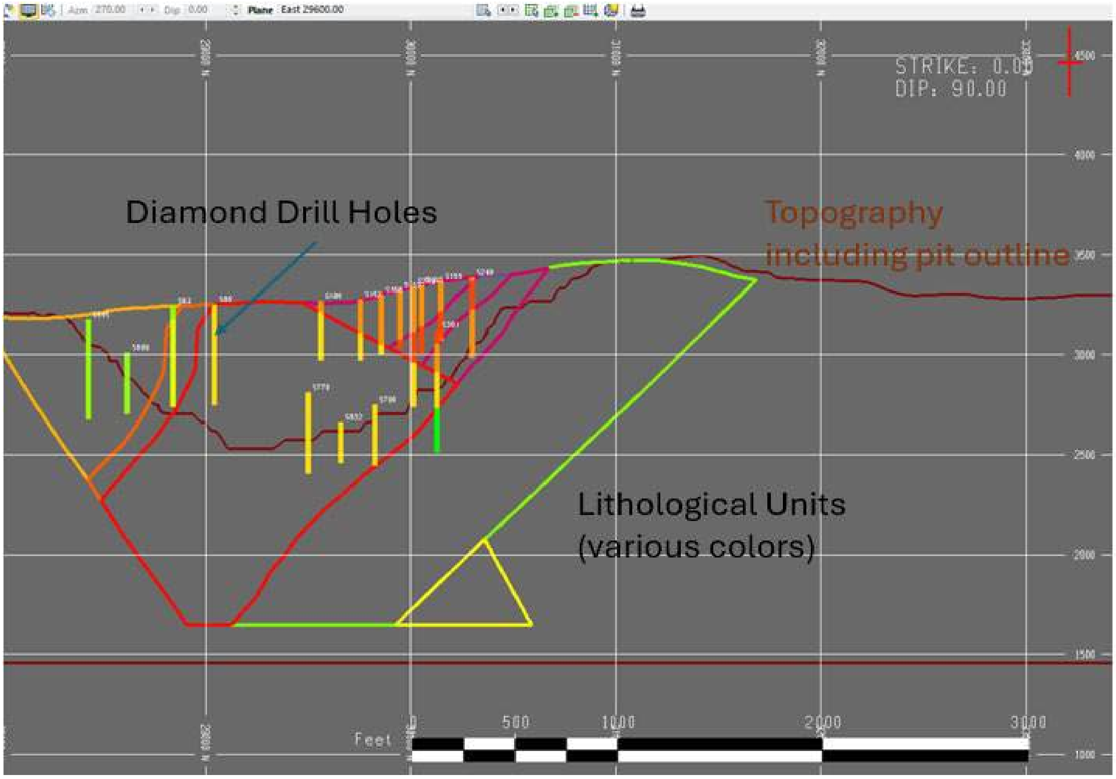


Figure 14.19 HxGN Mine Plan™ 3D Example of Drill Holes Flagged with Lithological Unit

Source: *AMPL, 2025*

14.5.4 Diamond Drill Hole Statistics

Statistics were run against each diamond drill group limited by lithology/lens/domain (see Figure 14.20 to Figure 14.33, below).

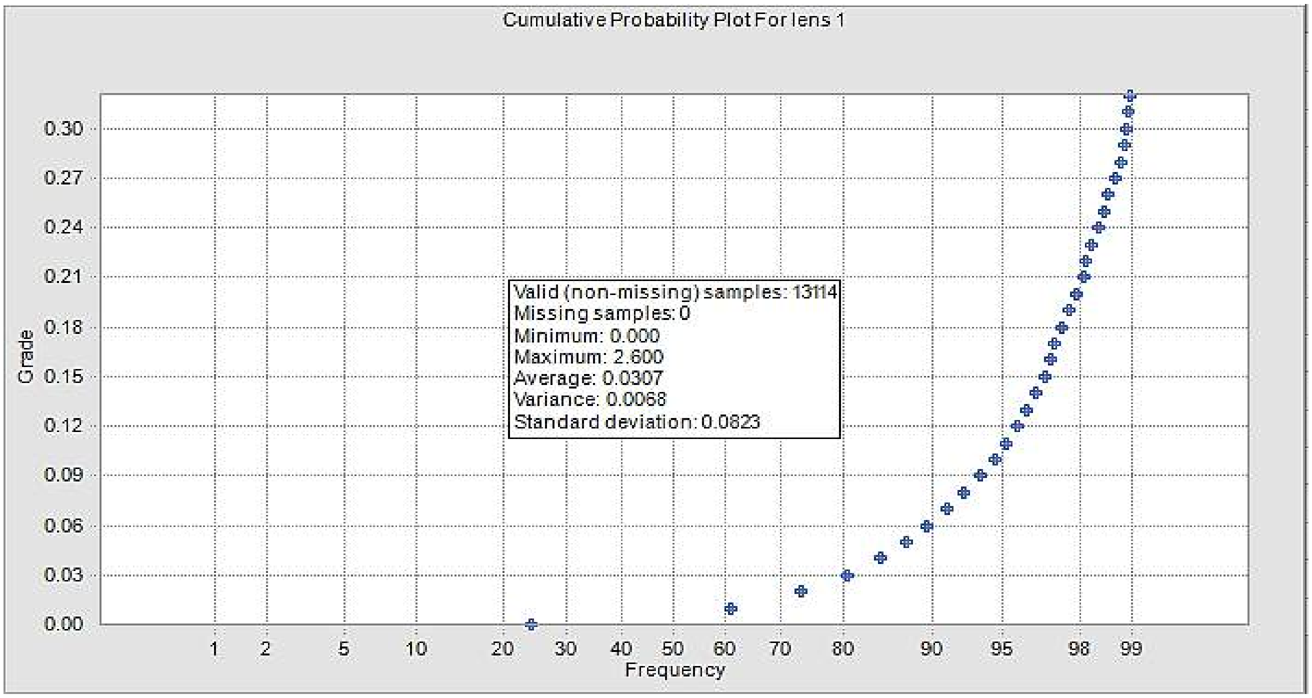


Figure 14.20 HXGnMinePlan™ Data Analyst Cumulative Probabliity Plot for Lens 1 (Denak NorthWest)

Source: *AMPL, 2025*

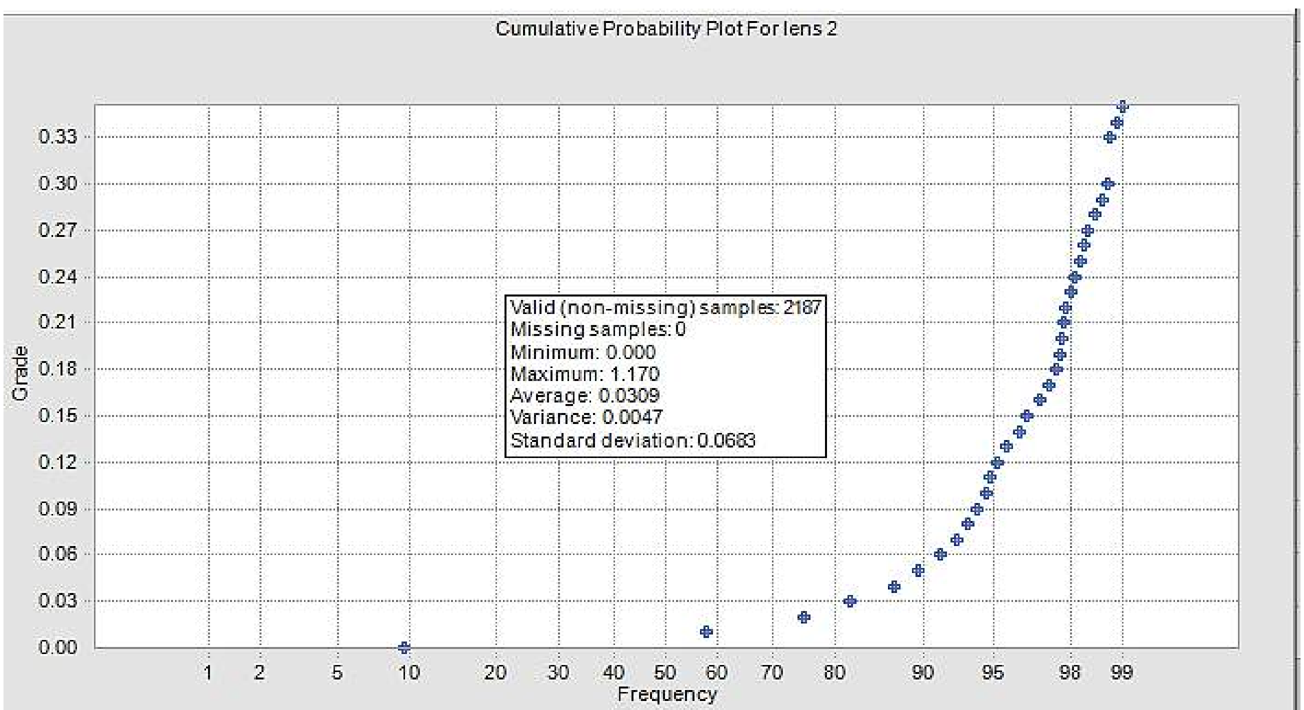


Figure 14.21 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 2 (Denak Pyrite Zone)

Source: *AMPL, 2025*

As expected, the pyrite zone shows greater variance based on unit, type of mineralogy and limited assays (see Figure 14.22 to Figure 14.25, below).

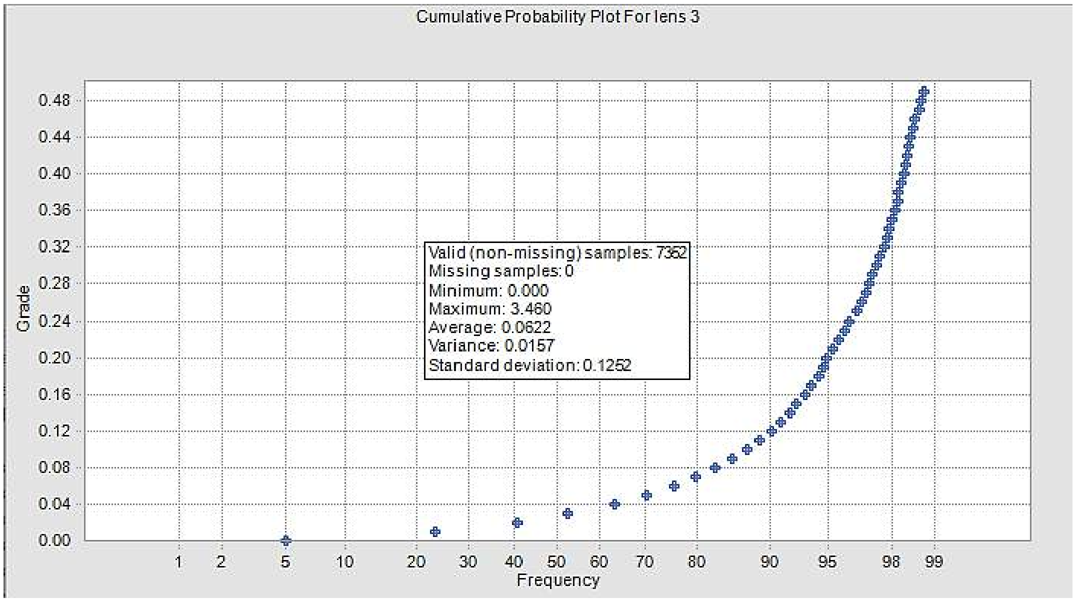


Figure 14.22 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 3 (Denak Sub East)

Source: *AMPL, 2025*

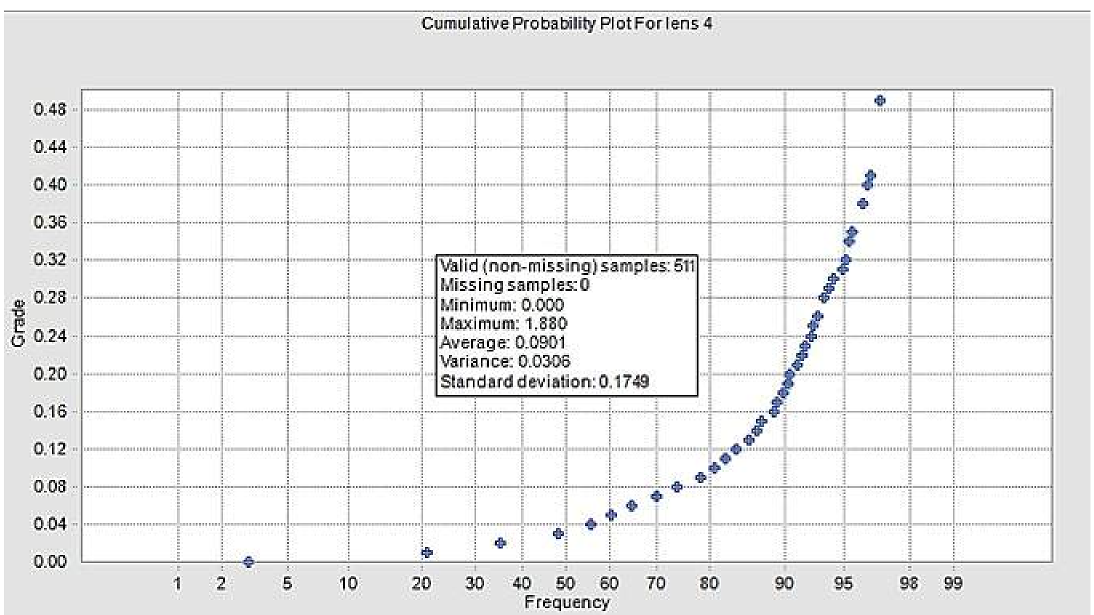


Figure 14.23 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 4 (Denak V-S06)

Source: *AMPL, 2025*

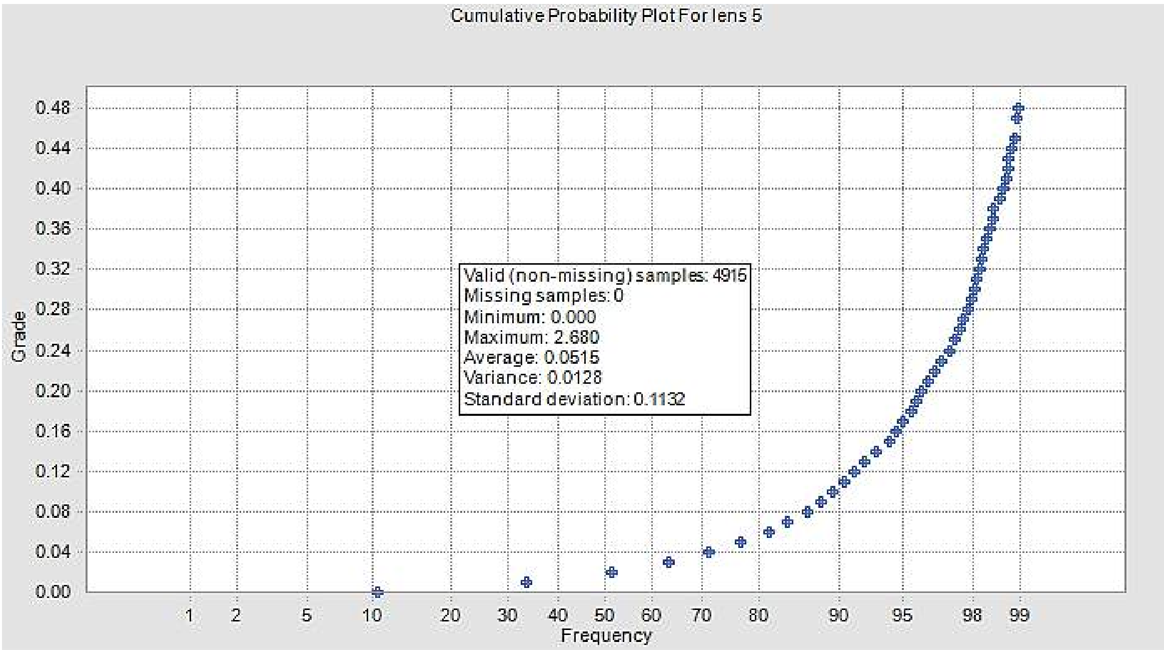


Figure 14.24 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 5 (Denak West)

Source: *AMPL, 2025*

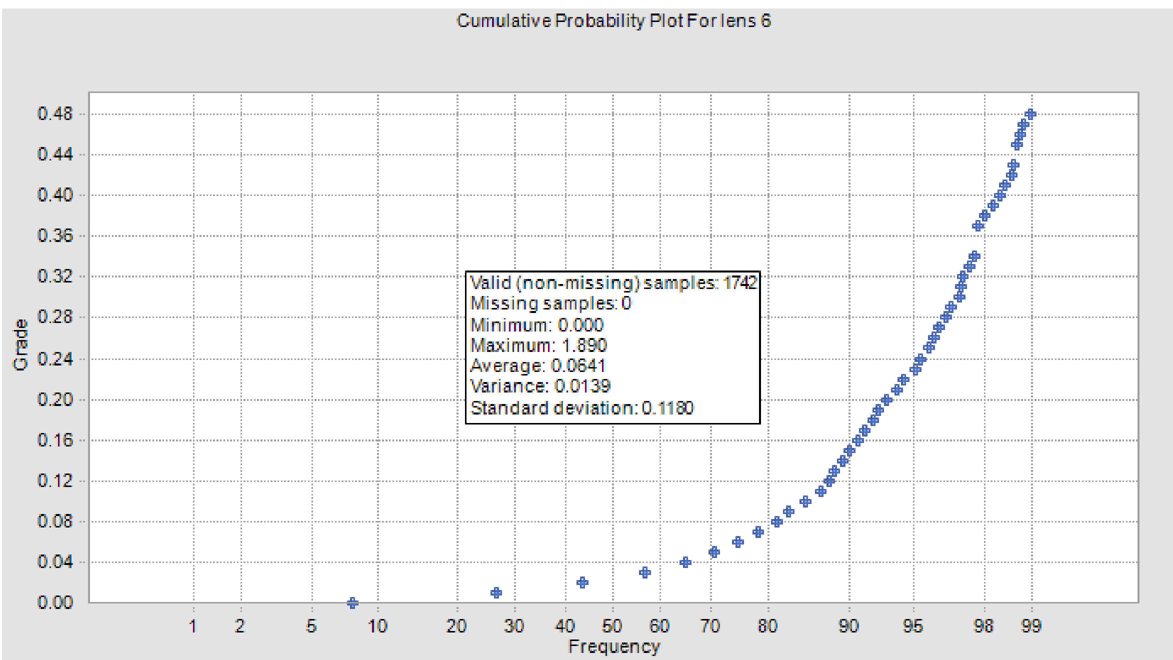


Figure 14.25 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 6 (Denak West Fault)

Source: *AMPL, 2025*

Surprisingly, the “fault zone” shows little unique variation (see Figure 14.26 to Figure 14.27, below).

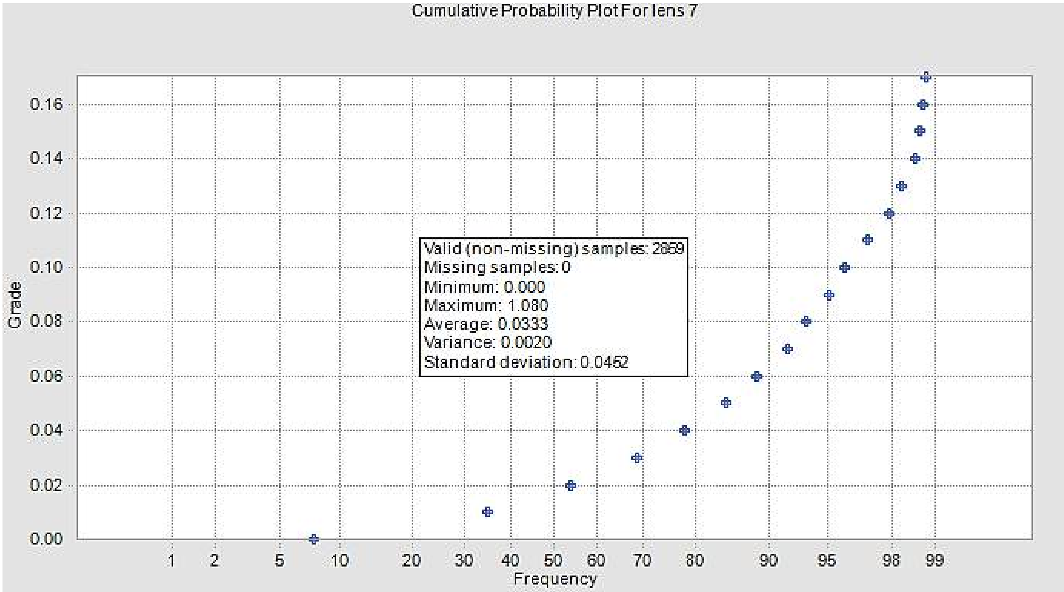


Figure 14.26   HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 7 (Endako K-Spar)

***Source:** AMPL, 2025*

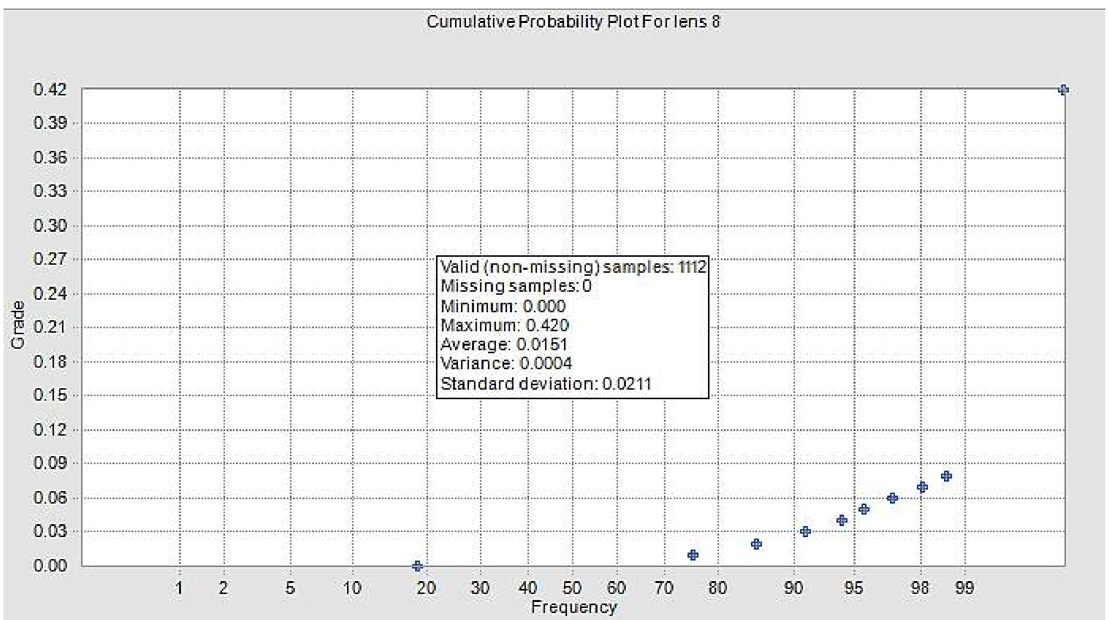


Figure 14.27   HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 8 (Endako Sub East)

***Source:** AMPL, 2025*

The graph is inconclusive due to the limited data (see Figure 14.28 to Figure 14.33, below).

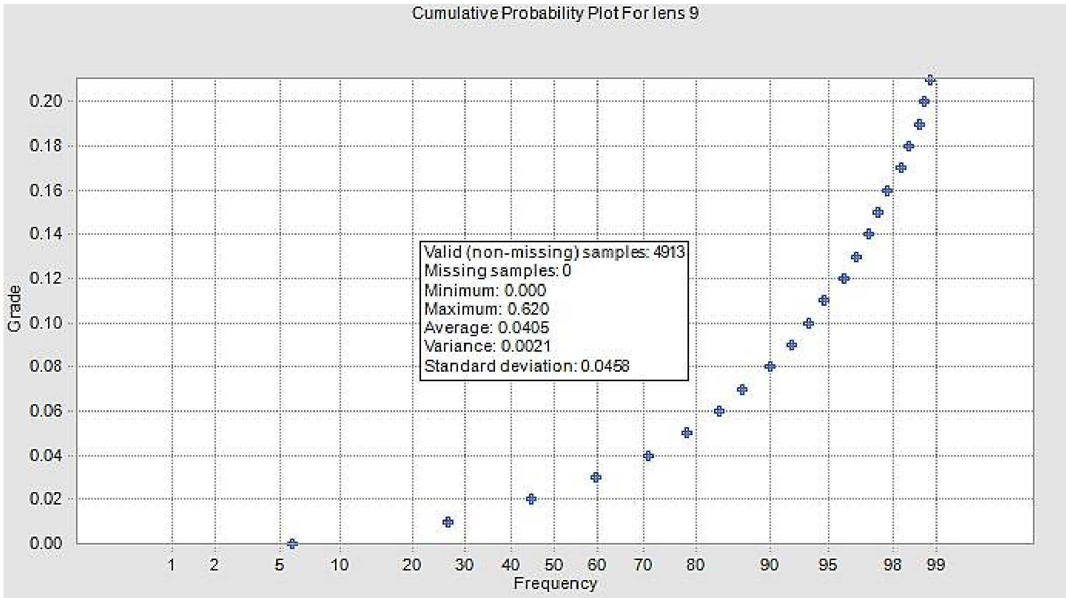


Figure 14.28 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 9 (Endako Pyrite)

Source: *AMPL, 2025*

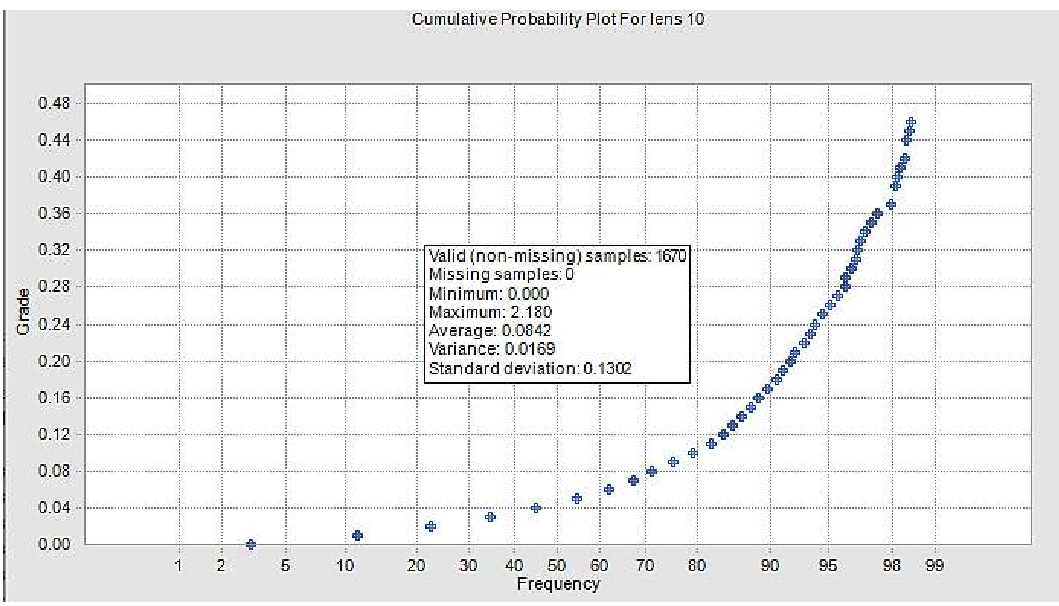


Figure 14.29 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 10 (Endako Sub-Basal)

Source: *AMPL, 2025*

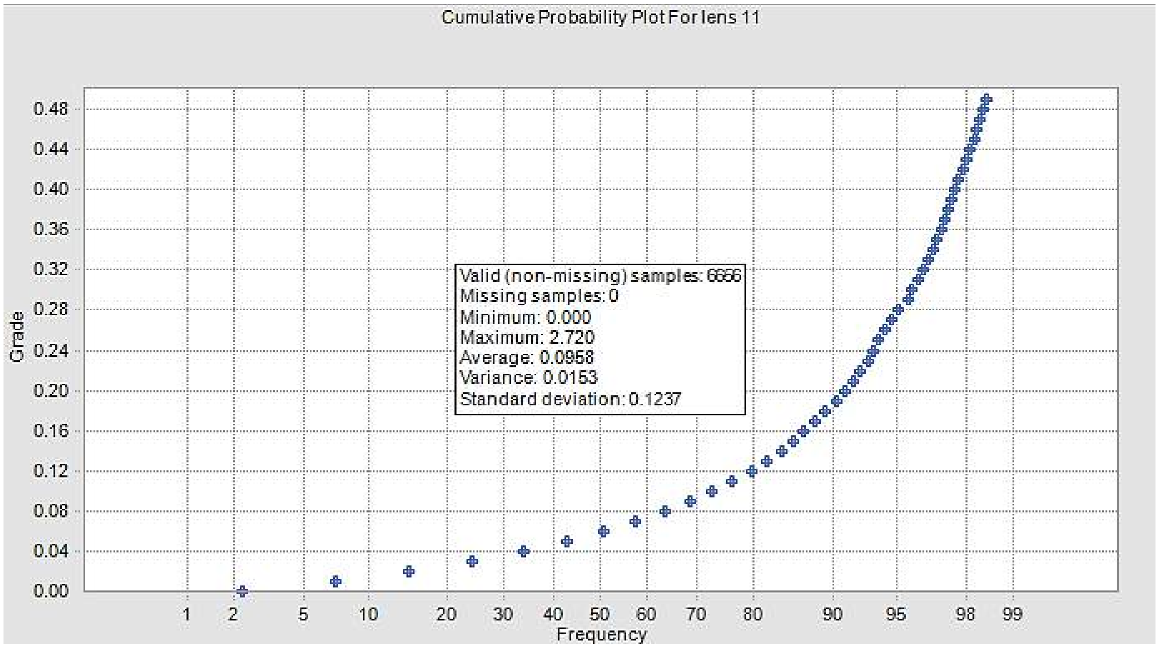


Figure 14.30 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 11 (Endako Sub East Extension)

Source: *AMPL, 2025*

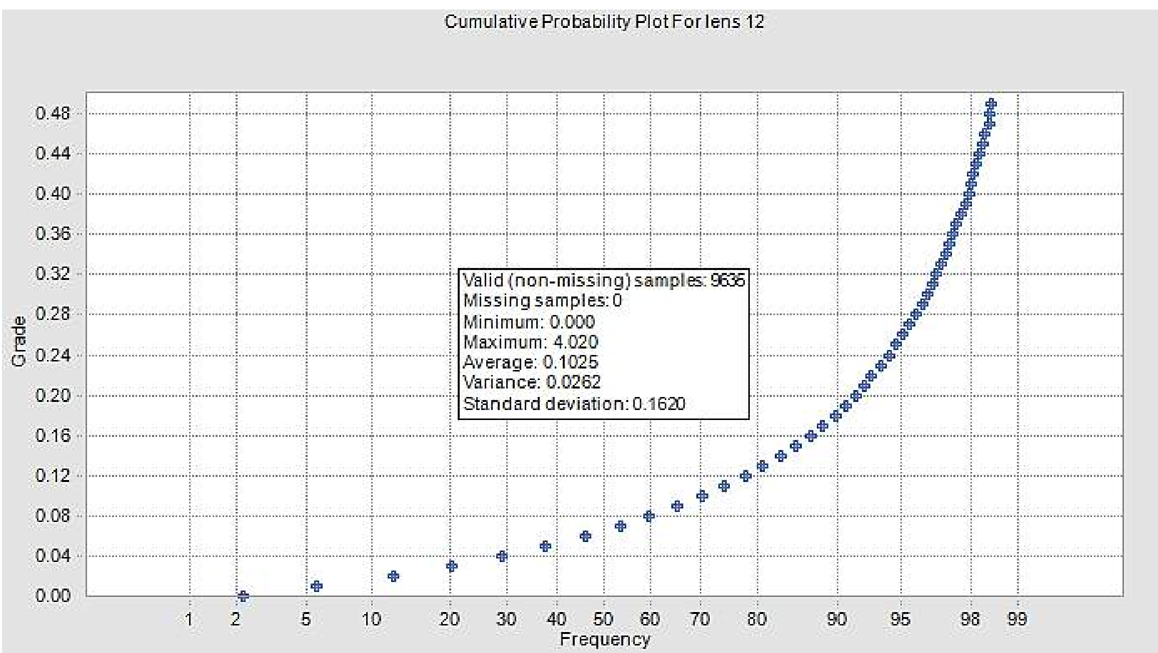


Figure 14.31 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 12 (Endako Sub West)

Source: *AMPL, 2025*

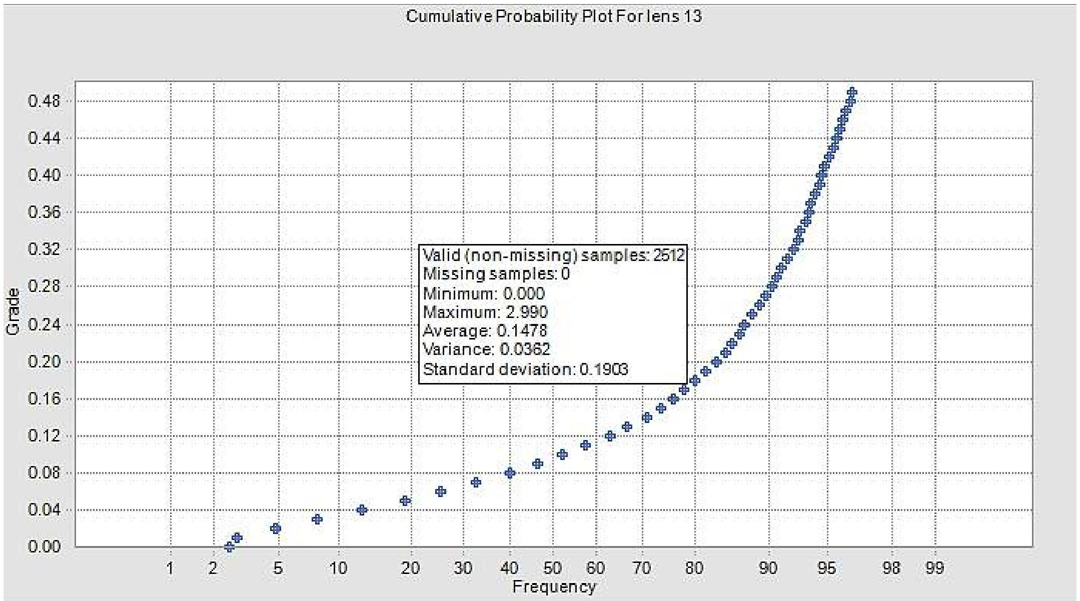


Figure 14.32 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 13 (Endako Vein)

Source: *AMPL, 2025*

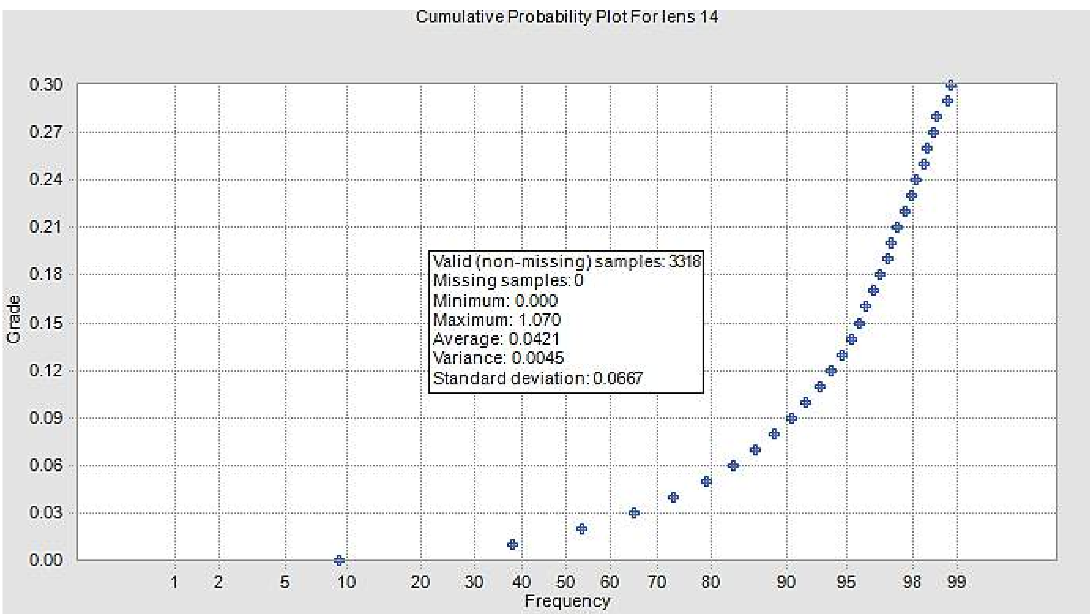


Figure 14.33 HXGnMinePlan™ Data Analyst Cumulative Probability Plot for Lens 14 (Casey)

Source: *AMPL, 2025*

14.5.5 Diamond Drill Hole Variograms

Variograms were created for each of the 14 lens/domains. The diamond drill assay file was simply chosen out of expediency. Since most of the assay file intervals are similar in length, it is the opinion of the QP that using the composited file would have made negligible difference. In addition, which is different from the norm, composite intervals were, in fact, generally shorter than the assay interval. This was done to match the 3D block parameters.

A total of several hundred planar two-dimensional (2D) variograms were created. However, due to space, only a couple are represented. They can be difficult and time consuming to interpret. HxGN Mine Plan™ 3D utilises a program MS3D™ Data Analyst that converts the 2D variograms into 3D variograms that can be incorporated into the software (see Figure 14.34, below).

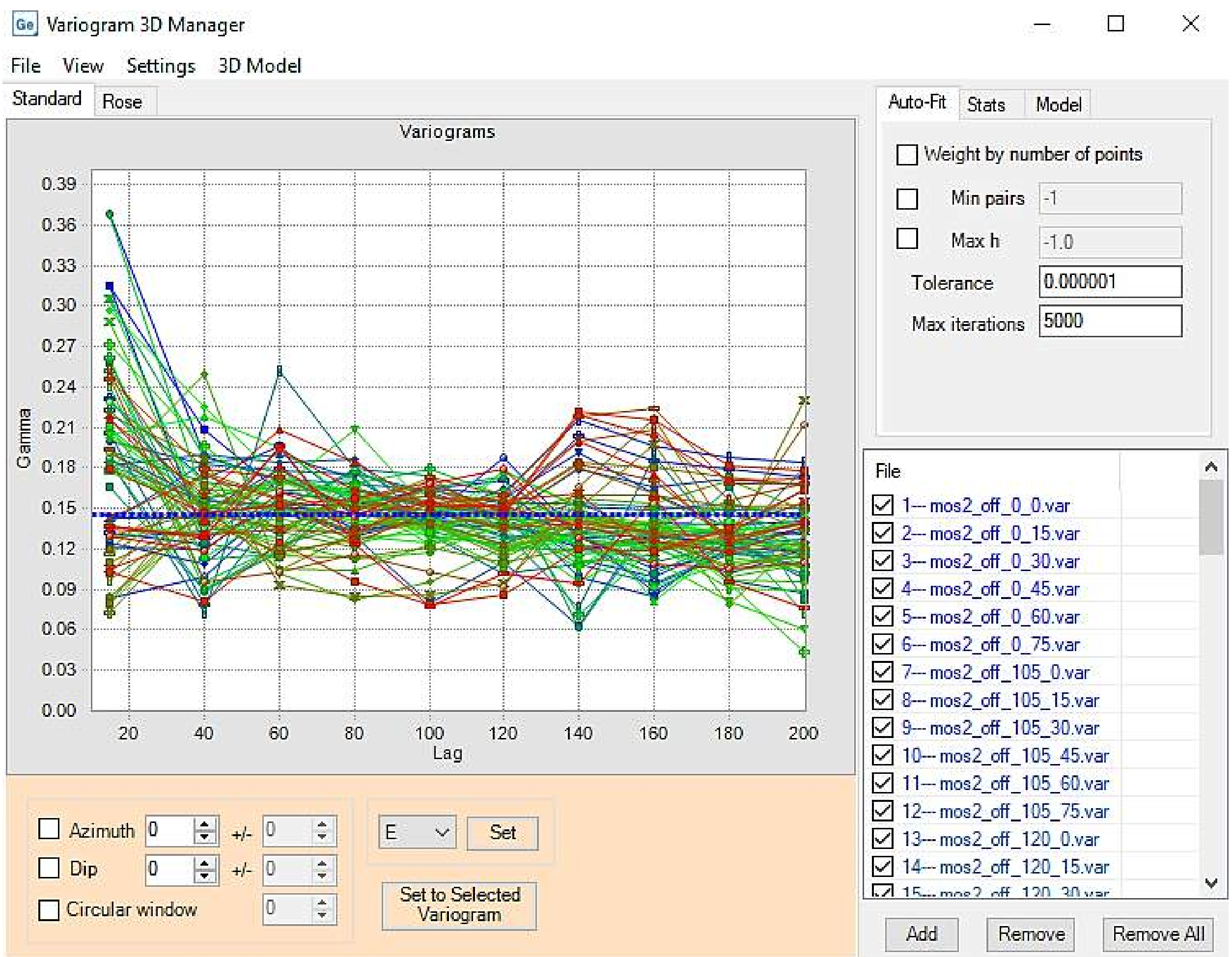


Figure 14.34 HXGnMinePlan™ Data Analyst Program Converting 2D Variograms into 3D Variograms
Source: *AMPL, 2025*

This was done with the variograms. If the reader has difficulty with interpreting 3D variograms, it is recommended that HxGN Mine Plan™ 3D/Hexagon can be contacted directly on their website (see Figure 14.35 to Figure 14.37, below).

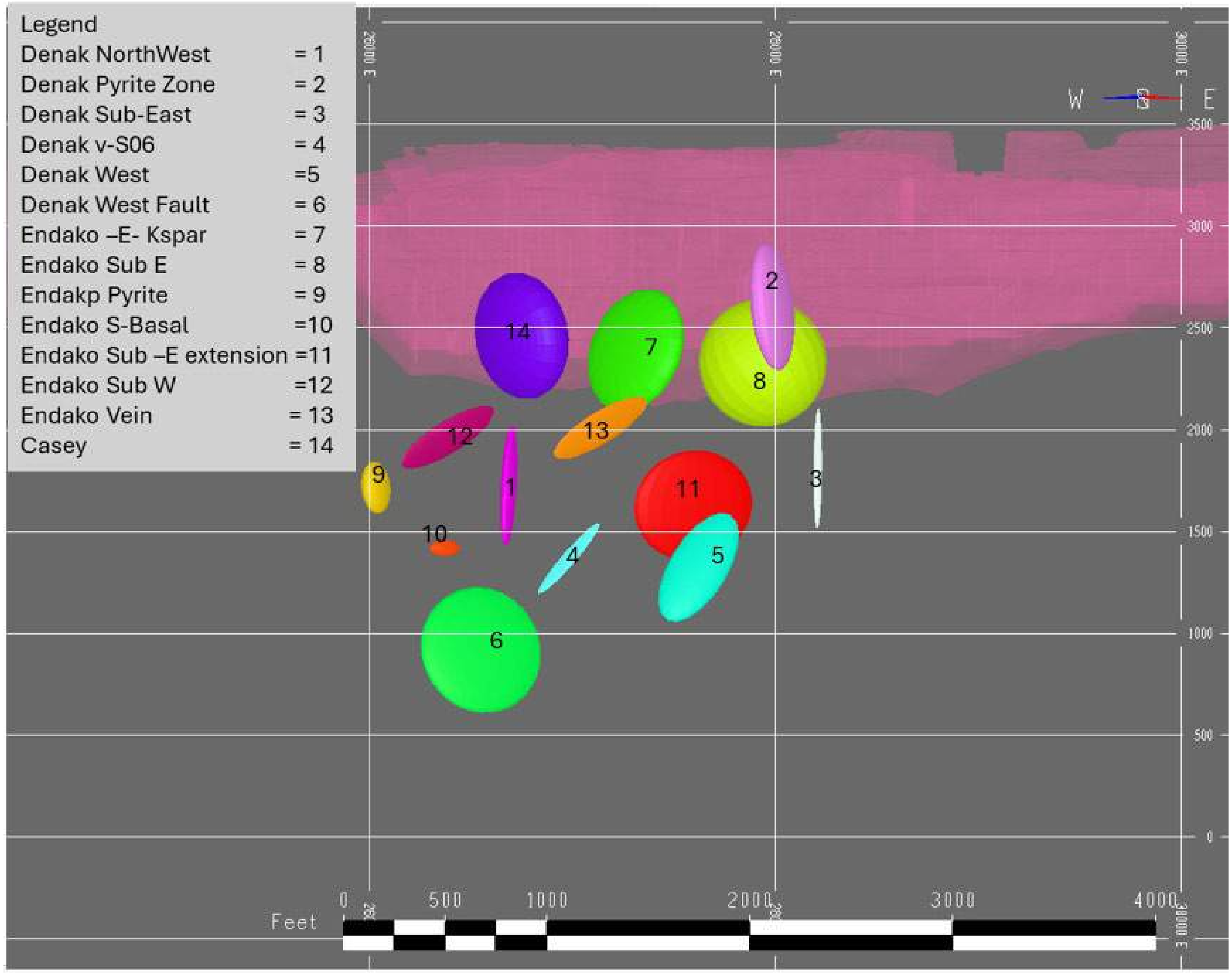


Figure 14.35 HxGN Mine Plan™ 3D Variograms Looking North
***Source:** AMPL, 2025*

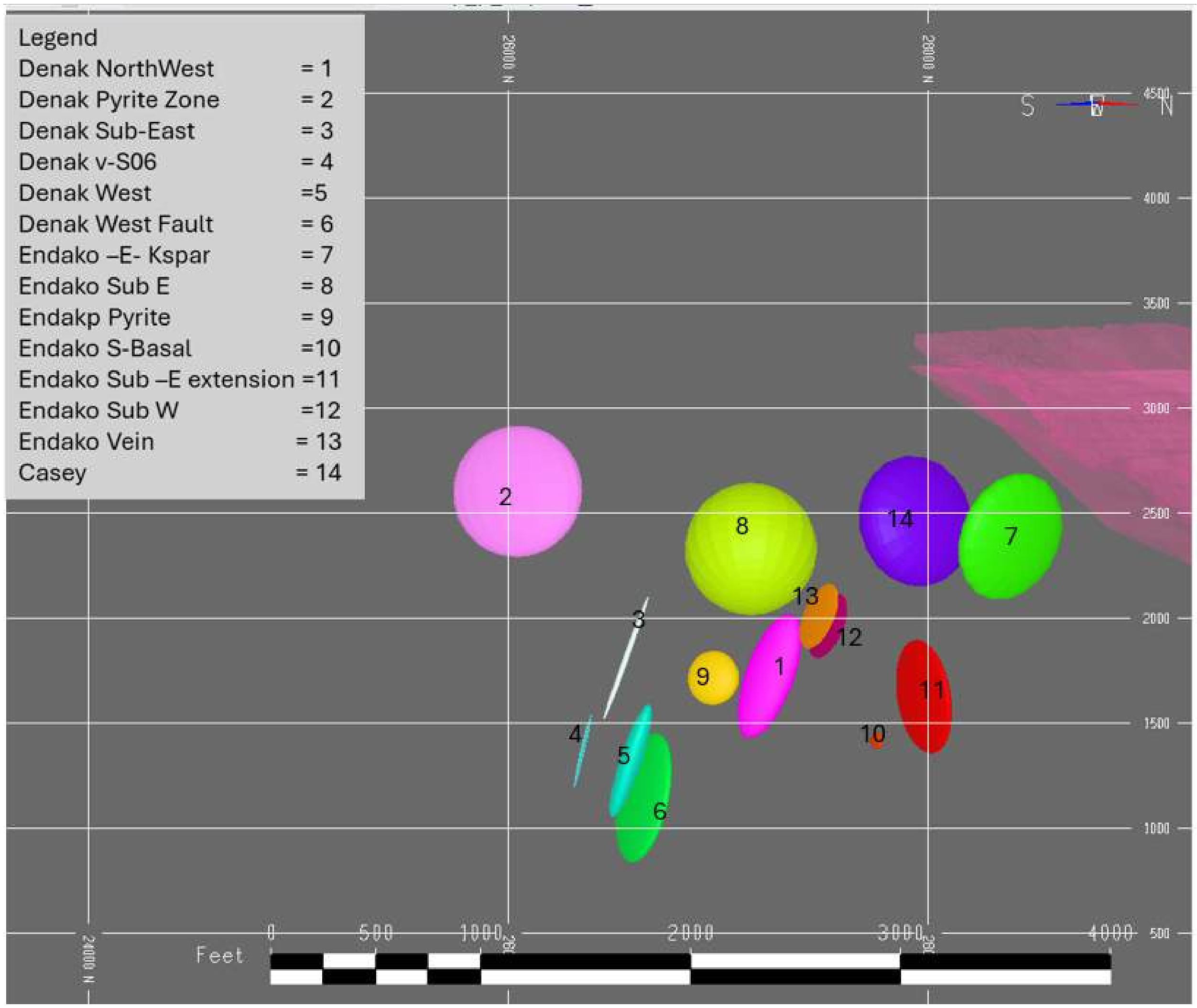


Figure 14.36 HxGN Mine Plan™ 3D Variograms Looking West
Source: *AMPL, 2025*

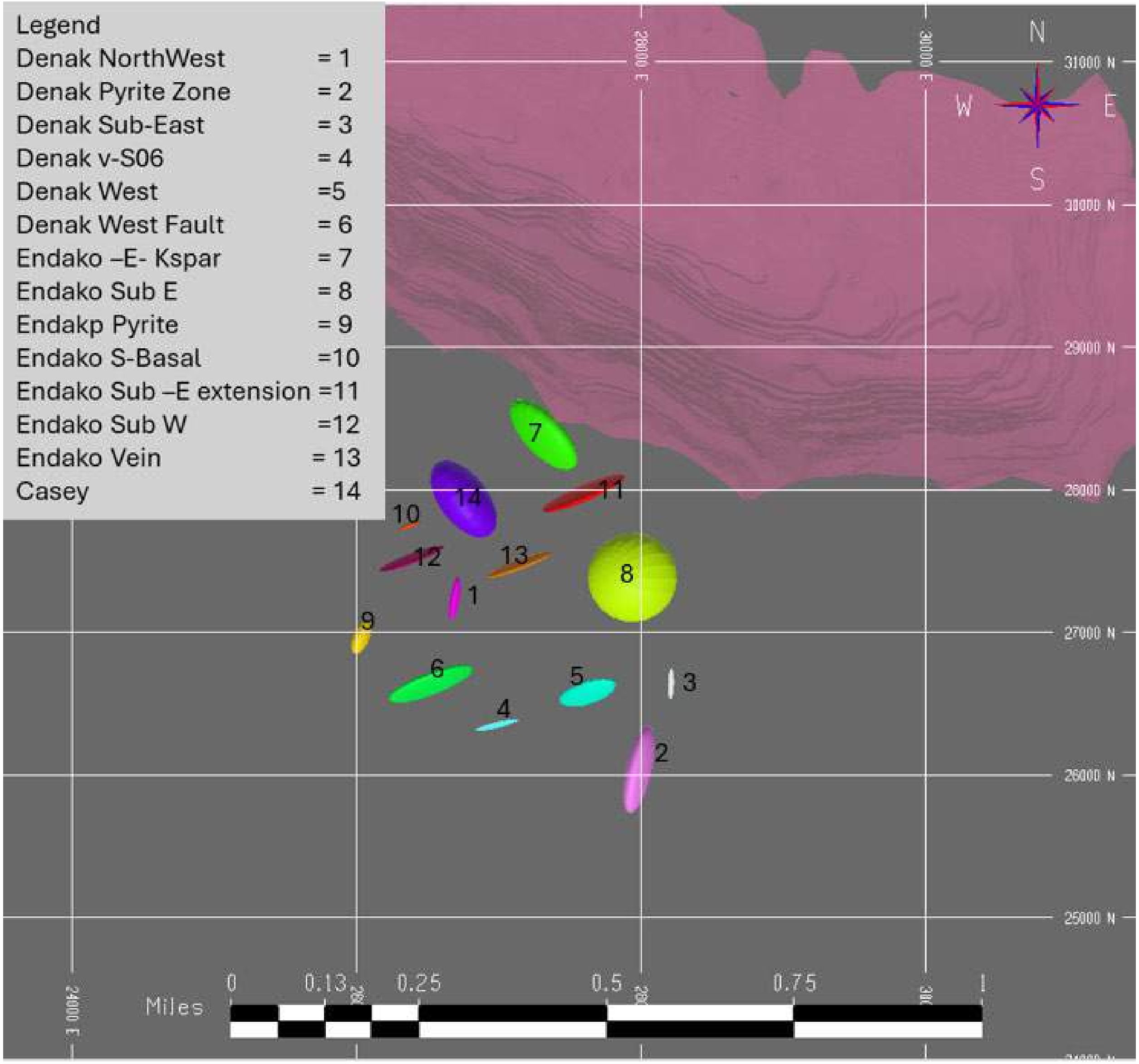


Figure 14.37 HxGN Mine Plan™ 3D Variograms Looking Down
Source: *AMPL, 2025*

14.5.6 Discussion on Variograms

Not surprising, a number of the variograms show discrete preferred orientations. Only the Endako Sub-E seems to have no preferred orientation and also shows the greatest continuity in all dimensions.

An approximate dimension of each variogram was then incorporated into the search ellipsoid for each of the lenses.

14.6 COMPOSITES

A generic composite interval of only 10 feet was chosen. Previous work had at times tried to match composites to bench height, but these have varied over time and much of the current resource is what could be best described as "salvage and expansion of existing pits". As such many proposed, the bench heights are unknown (see Figure 14.38, below).

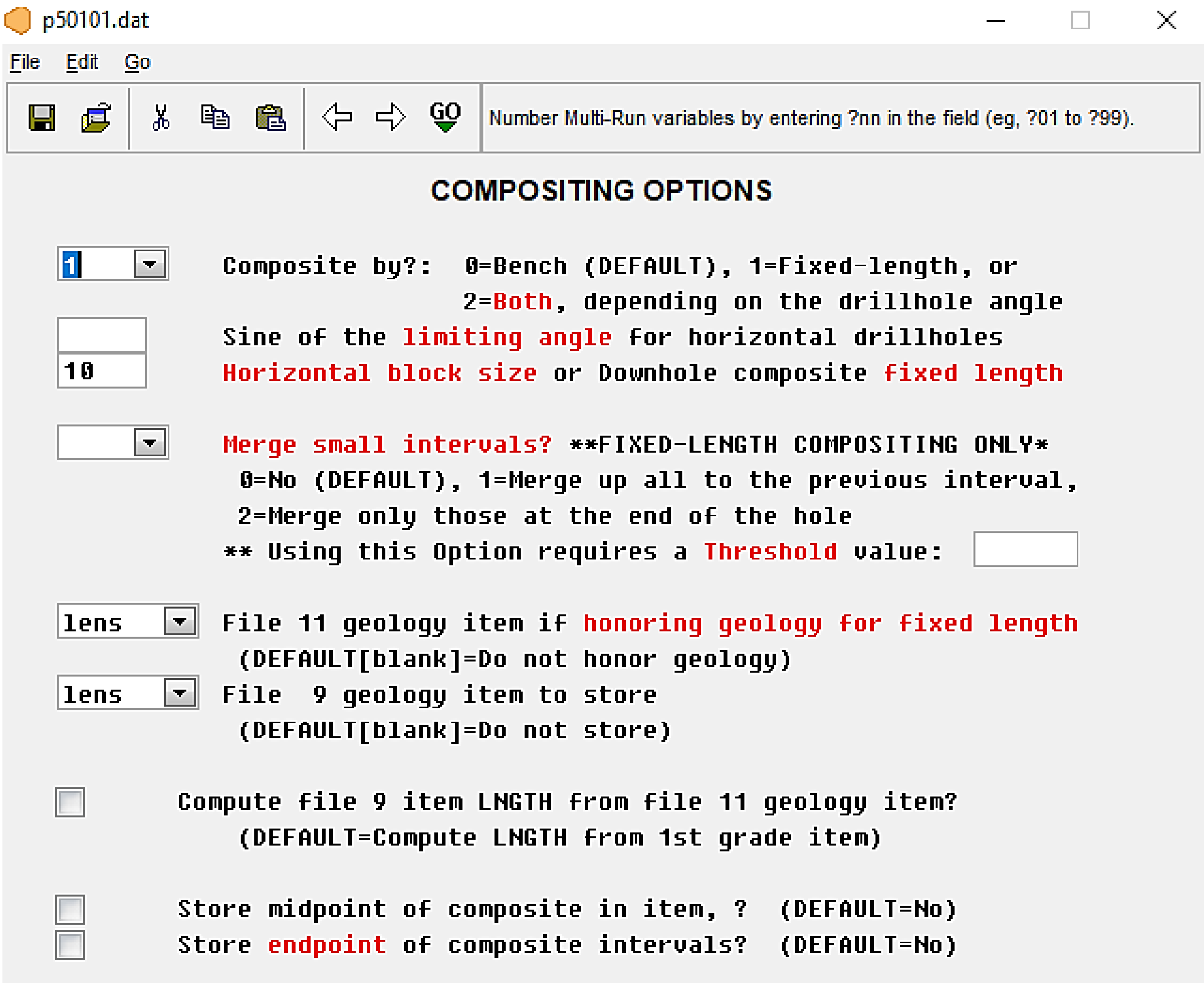


Figure 14.38 HxGN Mine Plan™ 3D Panel Showing How Composites Were Generated Using HxGN Mine Plan™ 3D
Source: *AMPL, 2025*

14.7 BLAST HOLES

Over 31,000 blast holes were also used in developing the block model. Previous work devoted considerable effort to declustering and determining statistics on these holes. In the opinion of the QP, this is not necessary and would have little or no impact on the Resource calculations. The holes are typically 20 ft apart from neighbouring holes, which would give a typical area of influence of 20 ft × 20 ft and a maximum depth of 25 ft or approximately 800 tonnes per blast hole. Because the material around the blast hole has obviously already been mined, it is only the influence that the blast hole has below the bottom of the hole, or holes in close proximity to the pit walls that can have an impact on the relatively local 25 × 25 × 25 search radius. If each blast hole represents 1,000 tonnes, then the maximum impact that the blast holes would have would be on 31 million tonnes or approximately 10% of the Resource.

However, it is still relevant that IMC states:

> *Recent production history (2008 and 2009, primarily Denak area) has shown that blast hole information agrees well with the reported mill head grades. Within that same volume, the diamond drill hole average grade was lower than the mill reported head by as much as 14%. Blast hole data in Denak reports higher than diamond drilling, as shown with the nearest neighbor comparisons at 20 ft spacing and with basic population statistics.*

The blast holes were then loaded in the same manner as the drill holes. They were kept in separate files and were run as separate programs at the end of the block model calculations. In other words, they overwrote the third pass of the drill hole program but only to a very limited extent as discussed above (see Figure 14.39, below).

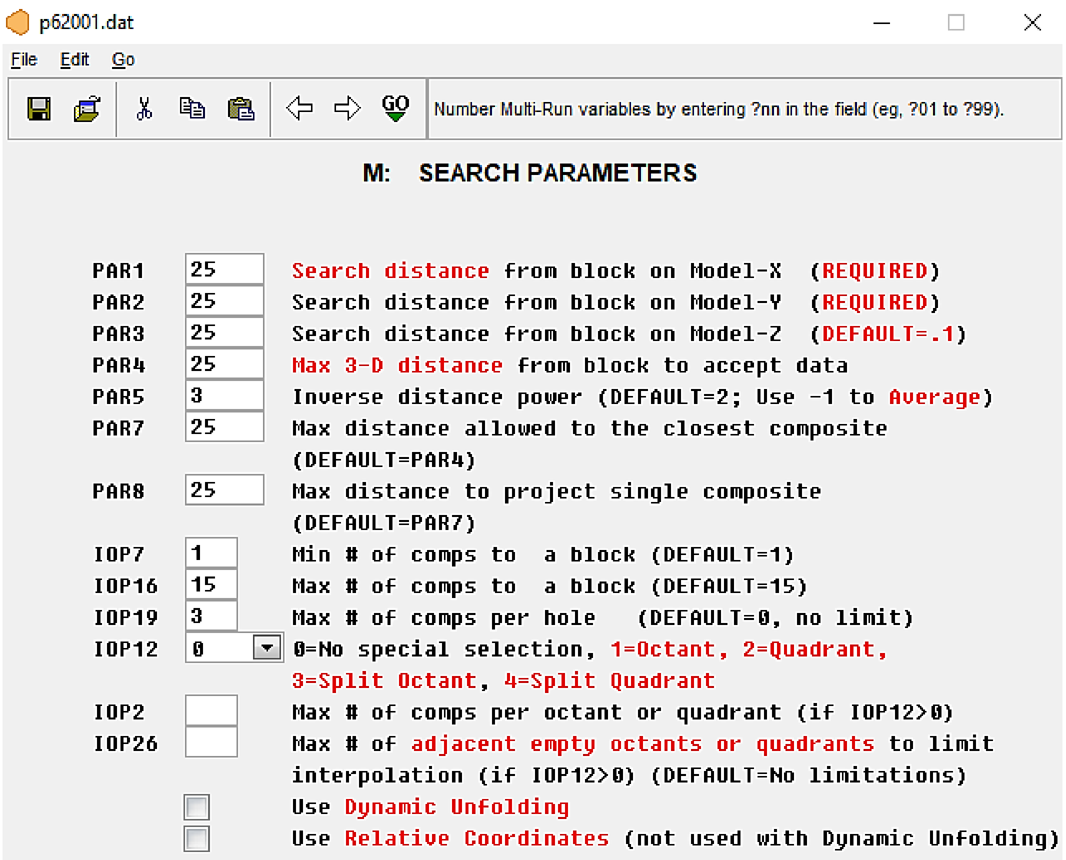


Figure 14.39 HxGN Mine Plan™ 3D Search Parameters Used on Blast Hole Data

Source: *AMPL, 2025*

14.8 3D BLOCK MODEL

The dimensions chosen were somewhat arbitrary. A bench elevation interval of 10 m was simply chosen since varying bench heights were employed by previous operators (see Figure 14.40, below).

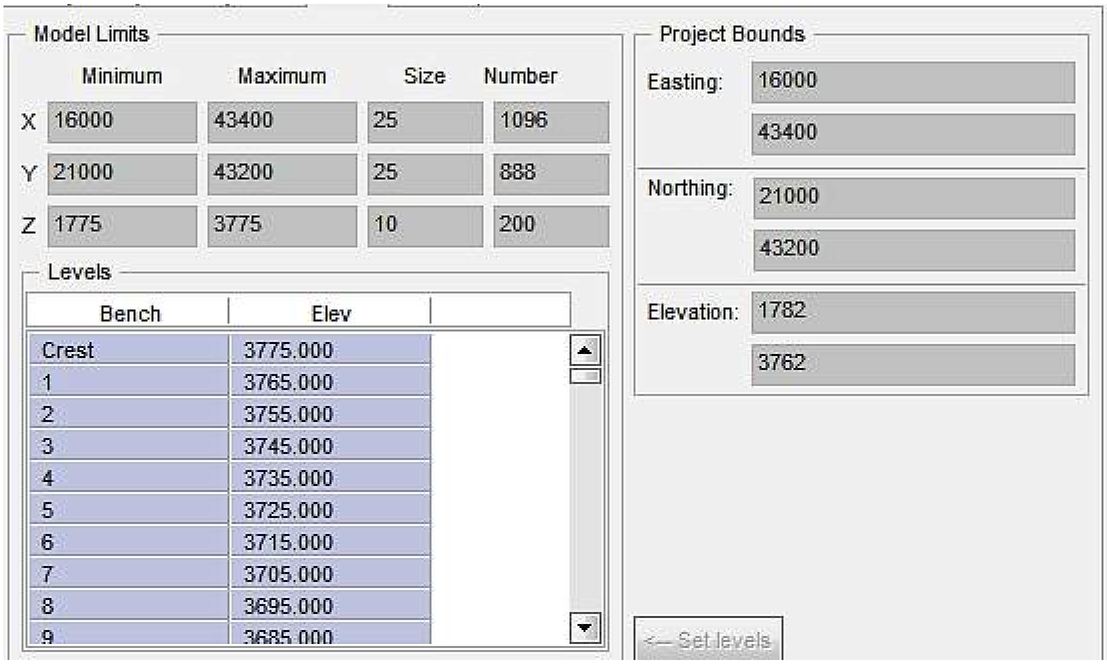


Figure 14.40 Panel Dimensions of 3D Block Model
Source: *AMPL, 2025*

Fourteen lens codes (domains) were created. Lens 1-13 material inside was based on previous geological modelling undertaken by the Endako staff. Lens 15 was material that fell outside the pit parameters. Where both lenses occupied the same block, preference was given to the material within the 1-13 domain.

A manual pit wall design roughly based on a 0.01% MoS_2 was used as the constraining limit of the model or in the case of the Casey, proximity to existing infrastructure (tailings pond) as well as honoring existing pit wall constraining limits. Because of the existence of an existing pit wall in most cases, the pit wall limits were often constrained by the existing wall rather than simply a grade cut-off (see Figure 14.41 to Figure 14.42, below).

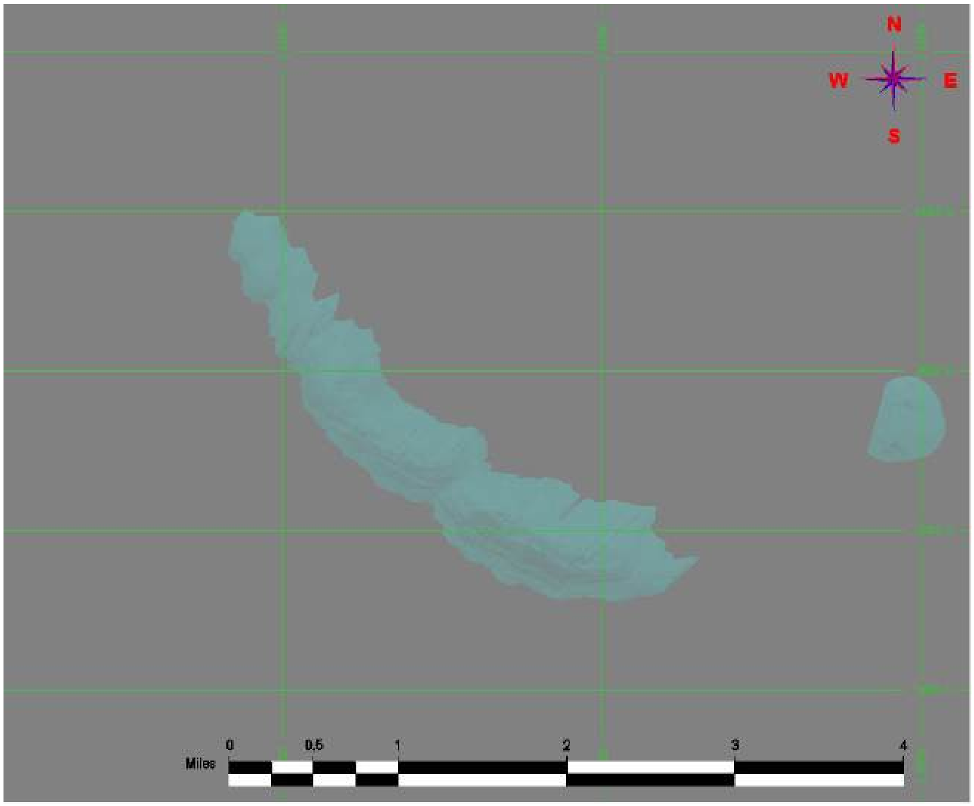


Figure 14.41 Plan View of Pit Constraints Based on Grade and Existing Pit
***Source:** AMPL, 2025*

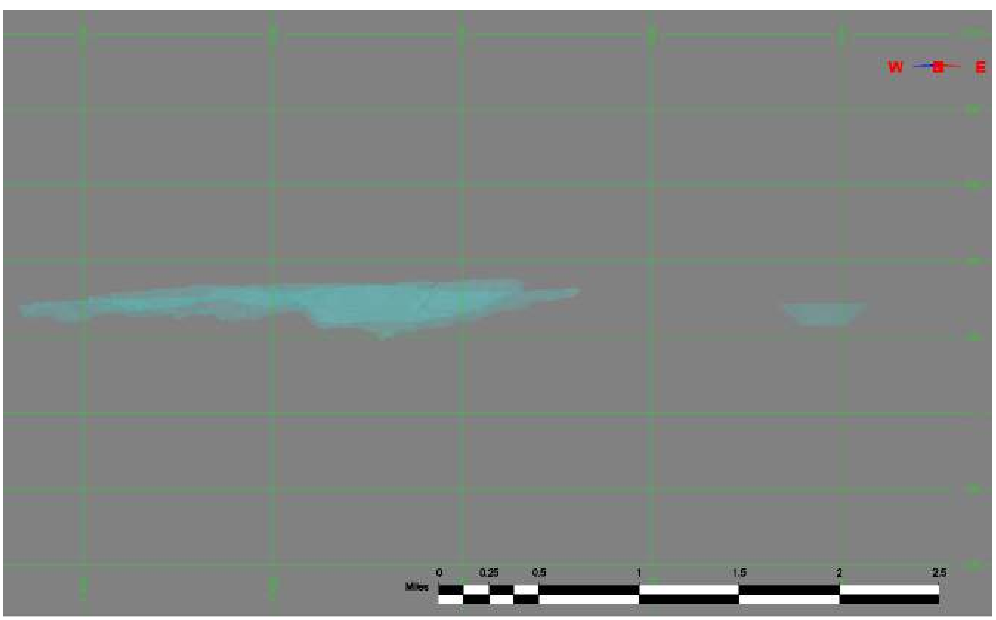


Figure 14.42 Sectional View Looking North of the Pit Constraints Based on Grade and Existing Pit
***Source:** AMPL, 2025*

The model was populated with the various lens codes/domains shown in Figure 14.43, below.

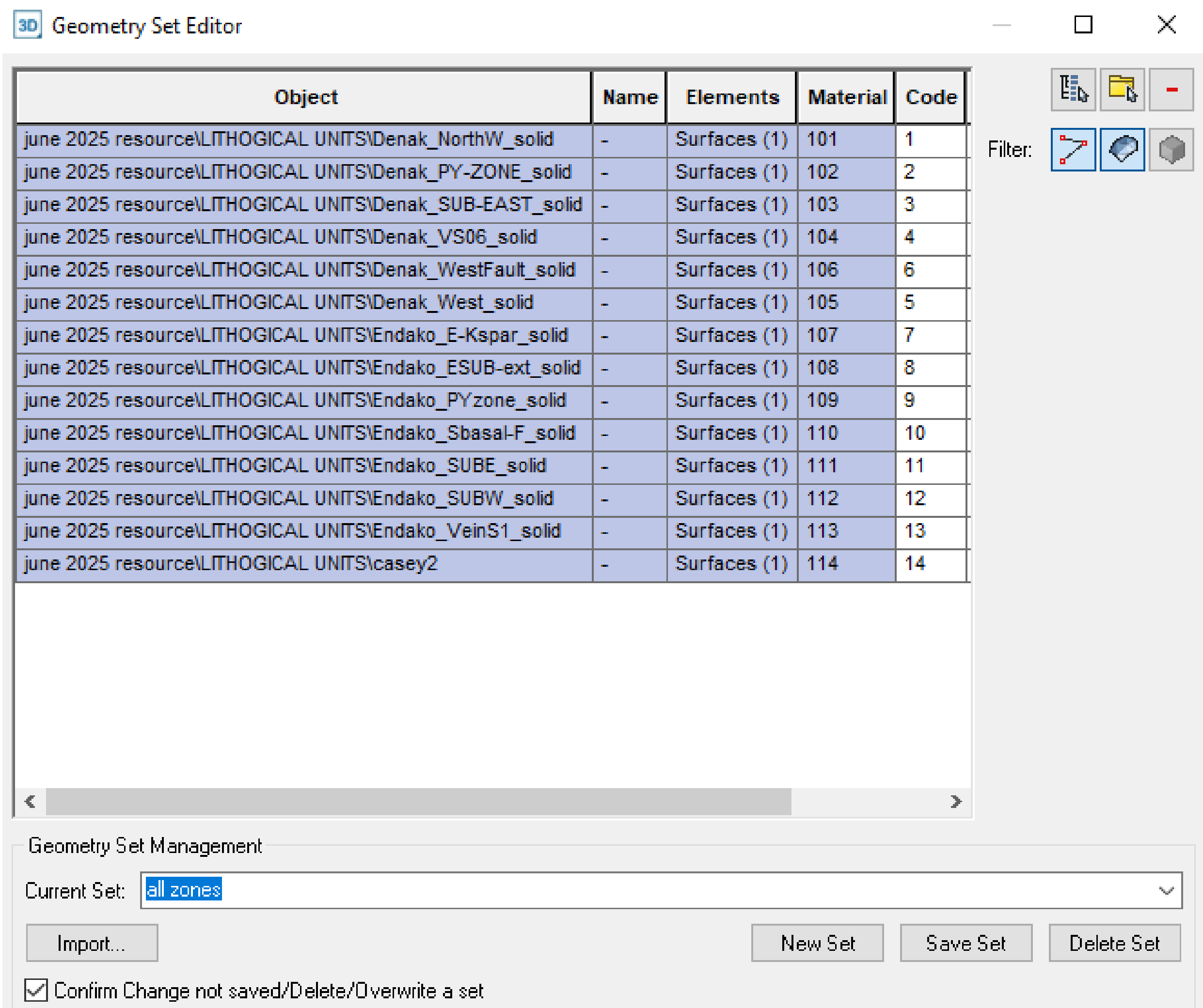


Figure 14.43 HxGN Mine Plan™3D Lens/Domains Assigned to Block Model
***Source:** AMPL, 2025*

Grades for MoS_2 were interpolated in three passes – the first pass would generally be used to calculate "Inferred Resource" in the search parameters (see Figure 14.44, below).

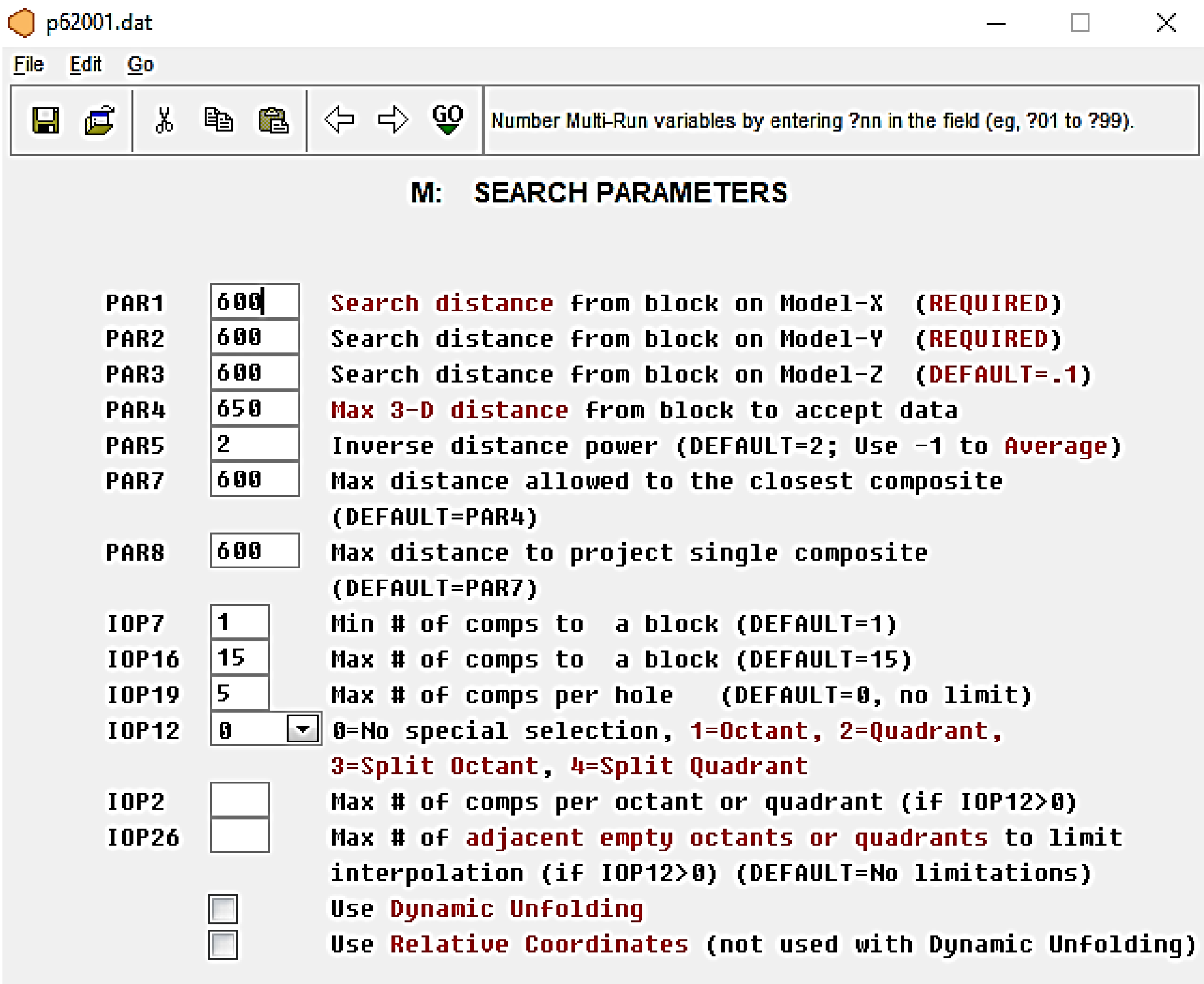


Figure 14.44 HxGN Mine Plan™ 3D "First Pass"
Source: *AMPL, 2025*

The same parameters were applied to all 14 domains. Due to the distances involved and the overall much smaller dimensions of the variograms, it was the opinion of the QP that attempting to control search parameters on this scale would not be relevant. It is important to note that while the original parameter states 0 to 600 ft, the subsequent second pass will overwrite the lower numbers. This is shown in Figure 14.45, below.

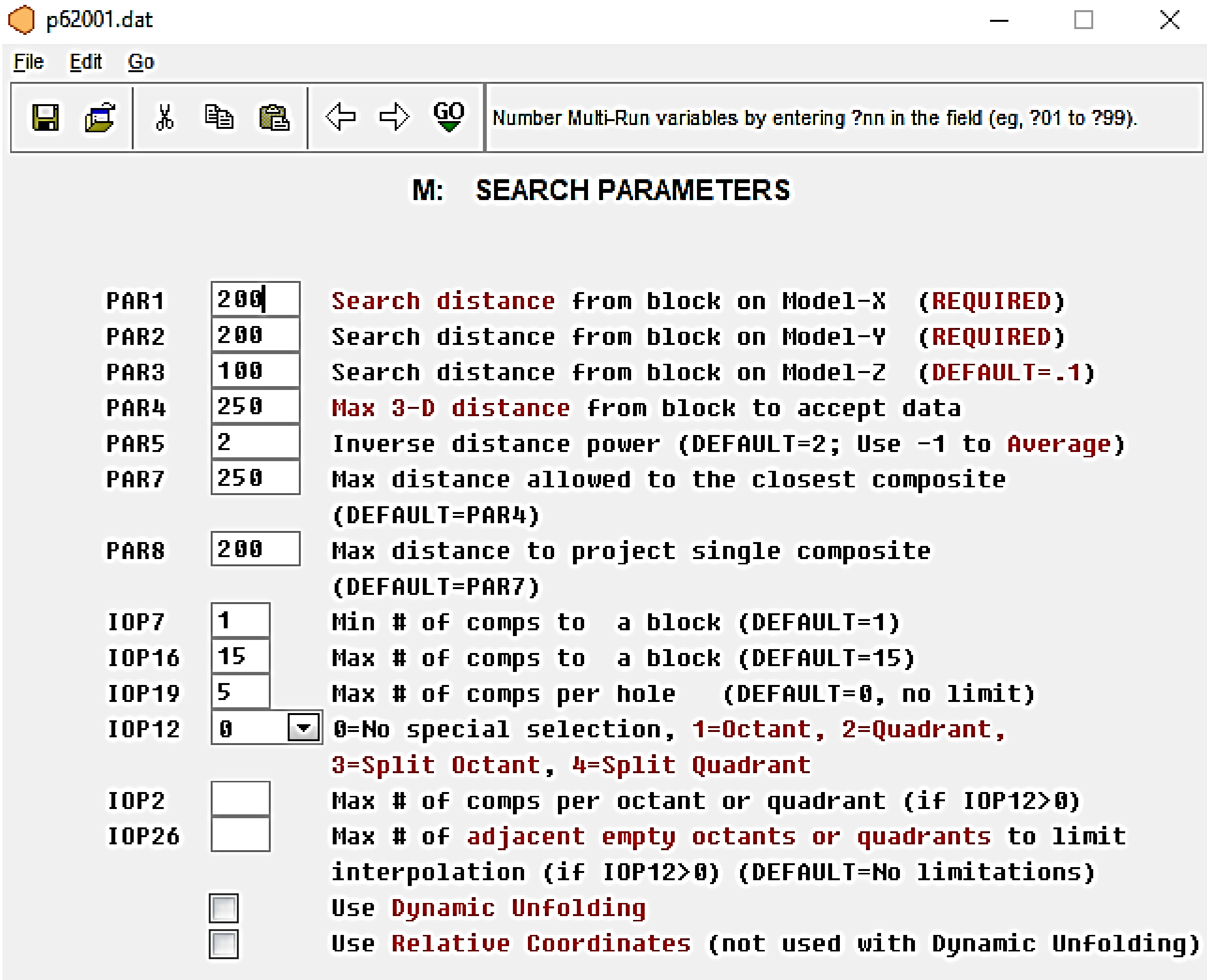


Figure 14.45 HxGN Mine Plan™ 3D "Second Pass"
***Source:** AMPL, 2025*

The second pass would generally be used to calculate "Indicated Resource". This pass overwrites the first pass out to 200, 200 and 100 ft, respectively. Again, this pass is generic in that it treats all 14 lens/domains the same (see Figure 14.46, below).

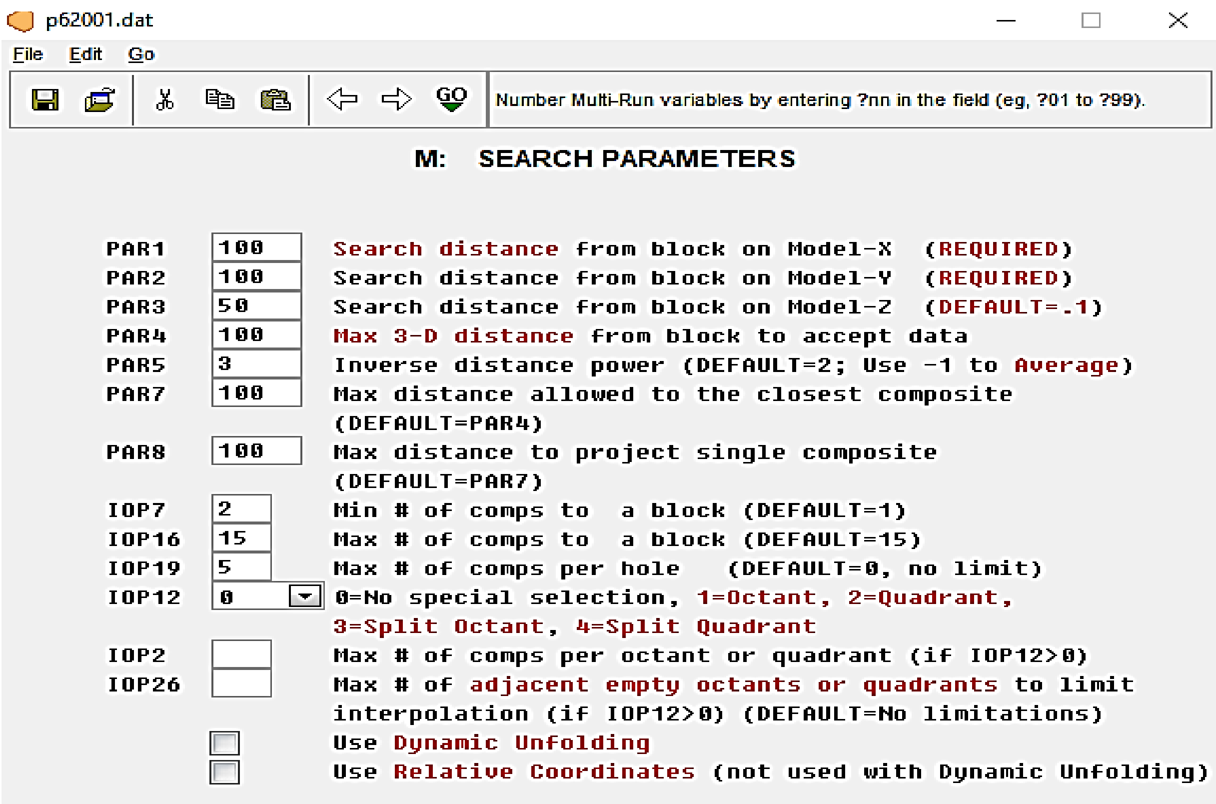


Figure 14.46 HxGN Mine Plan™ 3D"Third Pass"
***Source:** AMPL, 2025*

The third pass would generally be used to calculate "Measured Resource" – two hole minimum and ID^3. Specific variations for each of the 14 lens/domains were run. In other words, the third pass was, therefore, run a total of 14 times.

HxGN Mine Plan™ 3D overwrites each previous run file so only the most recent dimensions are retained, *i.e.*, 0-200 search ellipsoid will overwrite the portion of the 0-600 search used in the previous search.

Once the program was run for the diamond drill holes, it was once again run against the blast holes, which had also previously been flagged with lens code.

The program was then rerun with same the parameters for measured resource but with limited impact by the blastholes (*i.e.*, the search radius was limited to 50).

14.9 BULK DENSITY

Endako is unique in that linear measurement is Imperial units while rock mass is measured in metric tonnes and not in short tons. Production history is reportedly the primary support for the selection of density. Density measurements, compiled from a sub-set of the drill logs examined by AMPL, indicated that the drill core returned an in-situ density of 2.57, which is marginally higher than the density that was reportedly used at the mine site. It was noted that Endako discontinued measuring the density in drill core around 1978 or 1979. AMPL saw no reason not to change the numbers for in-situ rock as there is only a 0.27% difference in the numbers reported (see Table 14.8, below).

TABLE 14.8 DENSITIES REPORTED IN PREVIOUS REPORT	
In-situ Rock	13.781 cu ft/metric tonne (2.563 tonnes/cubic metre)
Overburden	19.619 cut ft/metric tonne (1.80 tonnes/cubic metre)
Broken Rock (*i.e.*, Waste Dumps)	17.657 cu ft/metric tonne (2.00 tonnes/cubic metre)
Tailings	19.619 cu ft/metric tonne (1.80 tonnes/cubic metre)

Source: *John M. Marek, P.E., 2011*

The only density that is overall relevant to this Resource estimate is the in-situ rock density of 13.781 cu ft/metric tonne or 0.0726 cu ft/metric per tonne used in the HxGN Mine Plan™ 3D software for estimation. Validation for the use of this density was given in Table 14.9, below. Some stockpile tonnes are included in the Mineral Resource and these use the 17.657 cu ft/metric per tonne value.

TABLE 14.9
COMPARISON OF DENSITY DATA ON SELECT DIAMOND DRILL HOLES

Hole ID	Count	Average Density	Zone
S341	47	2.56	Denak
S342	60	2.55	Denak
S346	61	2.56	Denak
S347	58	2.57	Denak
S348	47	2.55	Denak
S349	58	2.56	Denak
S350	69	2.55	Denak
S351	38	2.56	Denak
S354	56	2.56	Denak
S355	49	2.56	Denak
S356	56	2.57	Denak
S357	56	2.55	Denak
S358	60	2.57	Denak
S359	38	2.57	Denak
S412	41	2.58	Denak
S414	34	2.55	Denak
S415	44	2.61	Denak
S416	27	2.60	Denak
S462	5	2.59	Denak
S476	75	2.60	Denak
S541	28	2.63	Denak
S551	27	2.65	Denak
S384	50	2.58	Endako
S387	49	2.57	Endako
S405	51	2.56	Endako
S409	41	2.63	Endako
Total	**1,225**		
Median		**2.57**	

Source: *AMPL, 2025*

No recent bulk density measurements are available. Endako relied on the historical bulk density of 2.563 tonnes per cubic meter (t/cm^3), which is derived from production history. On the logs, density measurements were collected on every 10 ft sample. This practice was discontinued sometime between 1979 and 1980.

AMPL compiled density data on a sub-set of the drill logs. A total of 1,225 density values were extracted from 26 diamond drill holes. Results of this study indicated average density per hole ranging from 2.55 t/m^3 to 2.63 t/m^3 with an average of 2.57 t/m^3 (Table 14.9). The methodology used to measure the density of the drill core is unknown.

There is only limited data available for the Endako portion of the Resource, but the sub-set of 191 samples gave a slightly higher Specific Gravity/Density of 2.58.

14.10 CLASSIFICATION

Based on the study herein reported, delineated mineralisation of the Property is classified as a Resource according to the following definitions from National Instrument 43-101 and from CIM (2014).

> *In this Instrument, the terms "Mineral Resource", "Inferred Mineral Resource", "Indicated Mineral Resource" and "Measured Mineral Resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards (May 2014) on Mineral Resources and Mineral Reserves adopted by CIM Council, as those definitions may be amended.*

The terms Measured, Indicated and Inferred are defined by CIM (2014) as follows.

> *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. 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.*
>
> *The term Mineral Resource covers mineralisation and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction.*
>
> *The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing. 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 deposits,*

application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

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.

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 are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail 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.

Mineralisation 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 mineralisation. 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 Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

> *A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.*
>
> *Mineralisation or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralisation can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.*

Modifying Factors

> *Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.*

14.10.1 Reported Tonnages and Grade

A flow chart (Figure 14.47 and Figure 14.48, below) shows how the Resource estimate tonnage and grade were reported. From an initial gross in-situ tonnage of the pit design-controlled tonnage of over 1,015 million tonnes, the deposit was broken down into tonnage within 0.040% MoS_2. Using the calculations of “Measured,” “Indicated” and “Inferred,” as discussed above, the material was subsequently broken down into each of these categories based solely on the distance to a drill hole or blast hole.

Examination of the deposit (particularly in the area of Endako) indicated that while drill hole and blast hole density and surface exposure would qualify as much of the material as “Measured,” it was evident that there had been sloughing of material of unknown grade and tonnage into the existing pit and that the LIDAR images may not be absolute, most of the Endako pit was downgraded to Indicated category (or less). In addition, the Casey Zone was relegated to Inferred based on poorly understood geological constraints as well as the potential impact of the tailings dam design.

Figure 14.47 View of Endako Pit Showing Sloughing
Source: *AMPL, 2025*

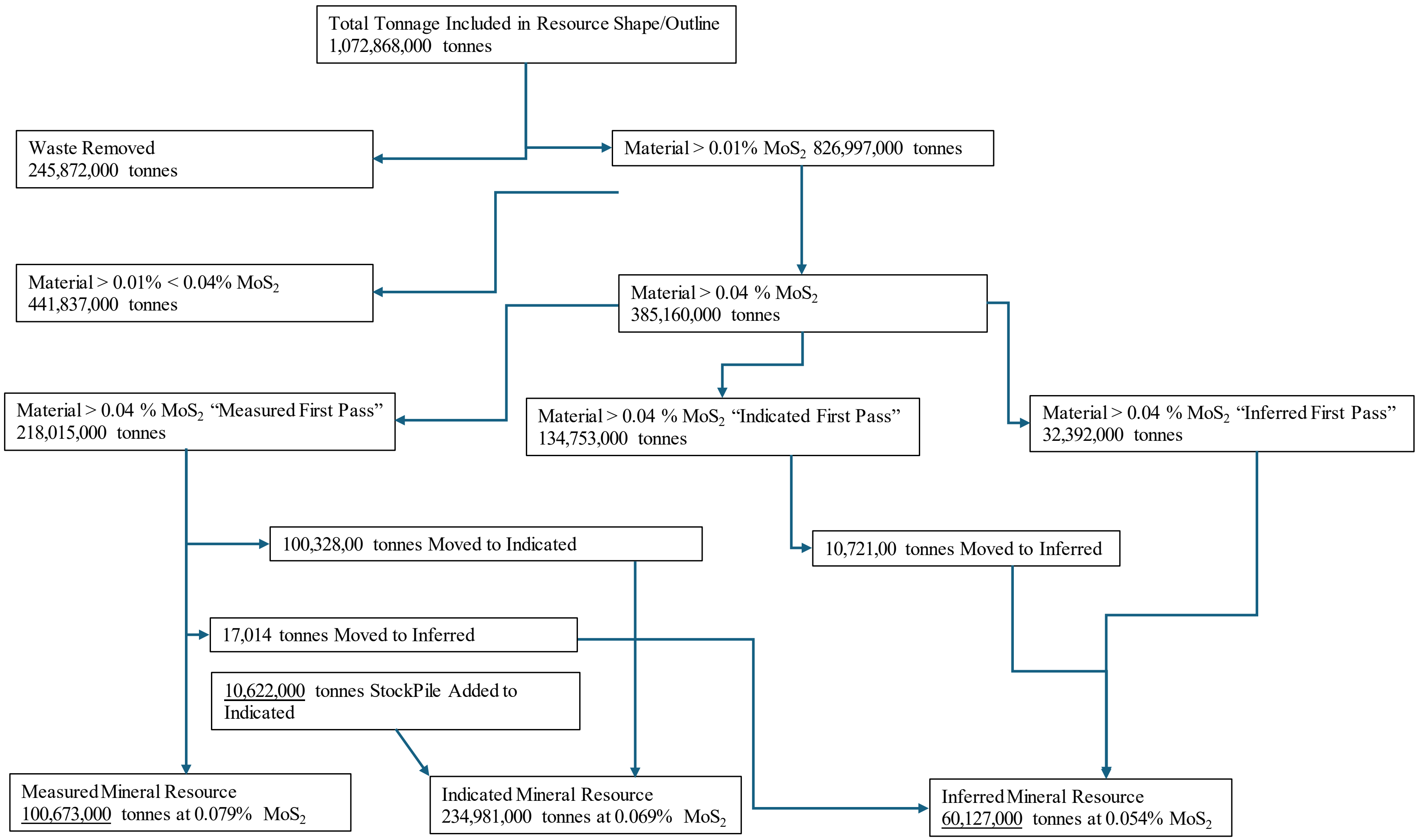


Figure 14.48 Flow Chart of Results
Source: *AMPL, 2025*

In addition, the Casey Zone was moved in its entirety to inferred. This was largely due to its close proximity to existing tailings dam enclosures, lack of detailed drilling and difficulty in designing an appropriate pit design.

14.10.2 Calculation of Cut-off Grade

Molybdenum has had a history of volatile pricing (see Figure 14.49, below).

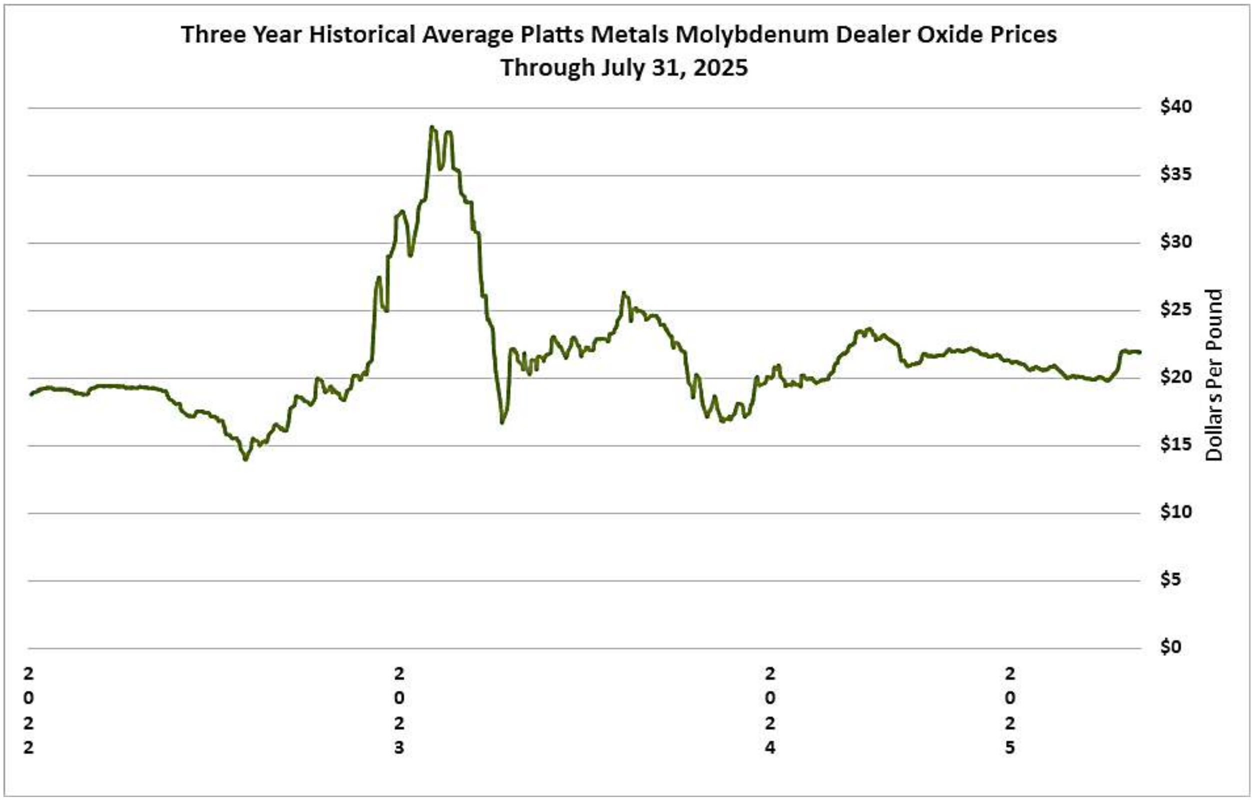


Figure 14.49 Molybdenum – 3 Year Prices

***Source**: Platts Metal is the data provider and Moon River Moly Ltd. is the designer of the graph, 2025*

The molybdenum contract is made available for trading with a unit of measure of dollars per pound. One contract unit represents 1,322.77 pounds, the equivalent of 60% of one metric tonne. The unit size was chosen to reflect standard trading terms in the physical molybdenum market, in which molybdenum oxide is priced on a "molybdenum contained" basis. Typically, the molybdenum content of molybdenum oxide is around 60%. This means that traders can trade the equivalent size in the futures size without having to adjust their futures trade for the "molybdenum contained" factor.

The estimated total average operating cost (excluding smelting and refining) is approximately $11.84 per tonne of potentially economic mineralisation. The operational costs given are for traditional mining methods. The 3-year average price of $22.50 per pound of molybdenum was use as well as and exchange rate of CA$:US$ = 1.35). Table 14.10, below, presents a summary of the assumptions used to calculate the break-even cut off grade for the resource of potentially economic mineralisation.

TABLE 14.10 ASSUMPTIONS USED FOR CALCULATION OF CUT-OFF GRADE	
Molybdenum Price per kg	$49.73
Molybdenum Price per lb	$22.50
% Molybdenum in MoS_2	59.94%
Mill Recovery	75.7%
Exchange Rate (CA$:US$)	1.35
All-In Operational Costs	$11.84

Source: *AMPL, 2025*

The breakeven cut-off grade required to meet the all-in operational cost of $11.84 per tonne is calculated as 0.039% MoS_2.

Breakeven Grade = Operating Cost (Price per kg × % Moly in MoS_2 × Mill Recovery × Exchange Rate)/1,000 = 0.039% MoS_2.

Using the parameters presented in Table 14.10, the calculated in-situ value at a cut-off grade of 0.04% MoS_2 is as follows:

Grade/100 × 1,000 kg/tonne × 59.94 (% Mo in MoS_2) × 0.758 % recovery/100 ×
US$49.59/kg × 1.35 = CA$12.17 In-situ Value

Based on the author's experience, a >0.040% MoS_2 cut-off was chosen as the in-situ value of this cut-off is just above the required breakeven grade.

14.11 BLOCK MODEL VALIDATION

The block model was verified for tonnage and grade using various HxGN Mine Plan™ functions.

1. Query function (essentially a spearing of solids routine).
2. PitRes™ – A Resource reporting tool in Hexagon.
3. UG1Res™ – A second Resource reporting tool using different parameters.
4. Individual lenses tonnage and grade were calculated and compared to the global resource

It is the opinion of the authors that the variances are acceptable. PitRes™ was used for all Resource calculations.

As indicated in Figure 14.50, below, the bulk of tonnage is in the 0.04% MoS_2 range. This matches reported tonnage and grade in the reported Resource at the aforementioned cut-off.

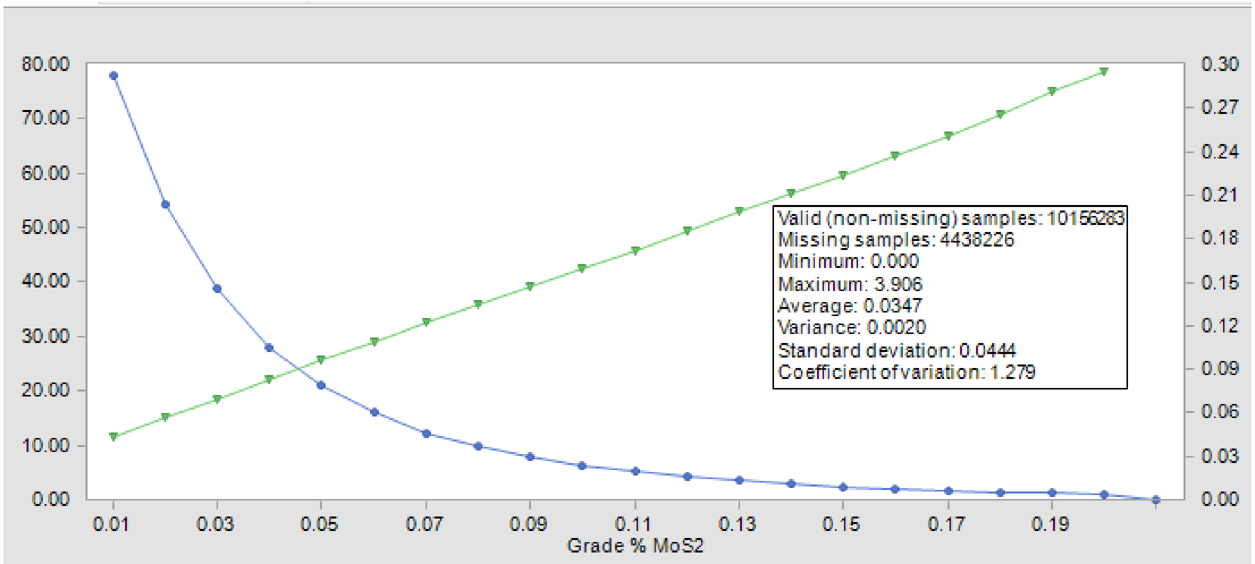


Figure 14.50 HxGN Mine Plan™ 3D Grade Frequency Curve of Lens 1 through 14

14.12 STOCKPILES

Previous authors have referred to surface stockpiles of both high-grade and low-grade mineralised material. However, there are a number of historical reports as well as wireframe models to validate the potential tonnage and grade (see Figure 14.51 and Figure 14.52, below).

Figure 14.51 "Ore" Stockpile
***Source:** AMPL, 2025*

Figure 14.52   "Ore" stockpile
***Source:** AMPL, 2025*

There are also physical manifestations of the stockpiles (see Figure 14.53 to Figure 14.56, below).

Figure 14.53   "Ore" and Waste Stockpiles
***Source:** AMPL, 2025*

Figure 14.54 "Ore" Stockpile
***Source:** AMPL, 2025*

Figure 14.55 "Ore" Stockpile'
***Source:** AMPL, 2025*

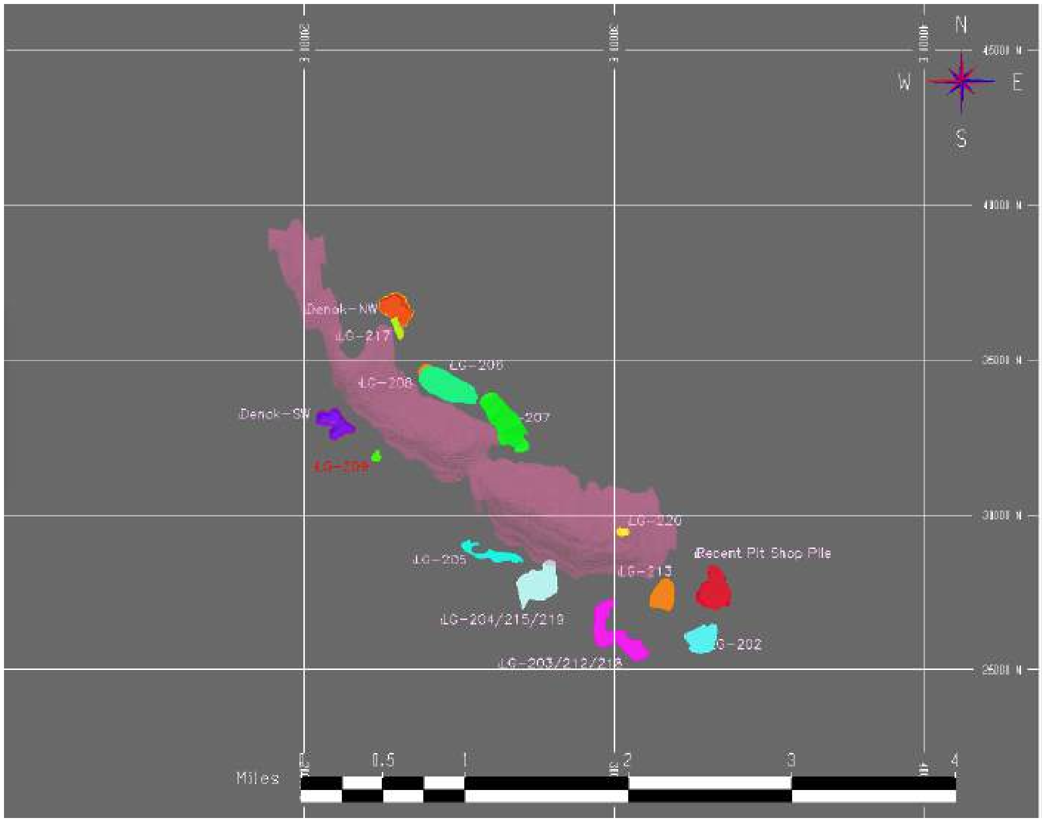


Figure 14.56 Purported Location of Stockpiles
Source: *Endako files – AMPL, 2025*

3D wireframe models were built based on location of stockpiles as reported in Endako internal reports, 2014 topography. It was impossible to determine with absolute certainty which stockpiles still existed and their grade. Based on wireframe modeling and using the density of broken rock, a theoretical potential tonnage was calculated in Table 14.11. below.

TABLE 14.11
TONNAGE AND GRADE OF STOCKPILES BASED ON 3D MODELLING

Stockpile	Cubic Feet		Tonnes	Estimate Grade – %MoS_2	
201		Mined Out			
202	28,242,602		1,599,513	0.075	From Reports
203	48,596,089		2,752,228	0.075	Estimated Average
204	63,363,696		3,588,588	0.075	Estimated Average
205	66,674,139		3,776,074	0.100	From Reports
206	21,485,589		1,216,831	0.078	From Reports
207	38,292,228		2,168,671	0.075	From Reports
208	-	Merged with 206	-		
209	921,286		52,177	0.075	From Reports
210	-	Mined Out	-		
211	-	Does Not Exist	-		
212	-	Merged with 203	-		
213	6,057,641		343,073	0.075	Estimated Average
214	-		-		
215	-	Merged with 204	-		
216	-	Mined Out	-		
217	-	Merged with Denak NorthWest	-		
218	-	Merged with 203	-		
219	-	Merged with 204	-		
220	-	Probably Gone in Edge of Pit	-		
221	-	Mined Out	-		
Denak NorthWest	42,794,184		2,423,638	0.075	From Reports
Denak Southwest	20,414,068		1,156,146	0.075	Estimated Average
Pit Shop	37,031,028		2,097,243	0.075	Estimated Average
Total	**373,872,550**		**21,174,183**	**0.080**	

Source: *AMPL, 2025*

Month end reports obtained for the last month of production did not mention the Denak stockpiles and indicated a lower tonnage. It was, therefore, decided to include the lower tonnage as reported by Endako in their month end summary. As a result, the stockpile was reduced to what was considered remaining in the 2014 Month End. Even so the tonnage and grade of the stockpile was conservatively reported as Inferred (see Table 14.12, below).

TABLE 14.12 TONNAGE AND GRADE OF STOCKPILES AS REPORTED IN THE ENDAKO MONTH END		
Stockpile	**Inferred Stockpile**	
	Tonnes	**% MoS_2**
202	1,600,000	0.075
203	1,500,000	0.082
204	3,589,000	0.080
205	459,000	0.076
Causeway	3,474,000	0.065
Total	**10,622,000**	**0.074**

Source: *AMPL, 2025*

- In-situ Rock.. 13.781 ft^3/metric tonne (2.563 t/cm^3)
- Overburden .... 19.619 ft^3/metric tonne (1.80 t/cm^3)
- Broken Rock (*i.e.*, Waste Dumps)................ 17.657 ft^3/metric tonne (2.00 t/cm^3)
- Tailings... 19.619 ft^3/metric tonne (1.80 t/cm^3)

While this tonnage was calculated using Lidar Surfaces and locations of stockpiles located from the Endako reports, it is uncertain whether some or much of the material was sent to the mill subsequent to the Lidar Survey. Instead, tonnage reported in the December 2014 Endako Month End is reported (see Table 14.13 to Table 14.34 and Figure 14.57 to Figure 14.58, below).

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

TABLE 14.13
HXGN MINE PLAN™ 3D STOCKPILES BASED ON 2014 LIDAR

Zone		Cut-off % MoS_2	Measured Resources		Indicated Resources		Measured and Indicated Resources		Inferred Resources	
			Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2
Stockpile	>=	0.010			10,622,000	0.074	10,622,000	0.074		
Stockpile	>=	0.015			10,622,000	0.074	10,622,000	0.074		
Stockpile	>=	0.020			10,622,000	0.074	10,622,000	0.074		
Stockpile	>=	0.025			10,622,000	0.074	10,622,000	0.074		
Stockpile	>=	0.030			10,622,000	0.074	10,622,000	0.074		
Stockpile	>=	0.035			10,622,000	0.074	10,622,000	0.074		
Stockpile	>=	0.040			10,622,000	0.074	10,622,000	0.074		
Stockpile	>=	0.045								
Stockpile	>=	0.050								
Stockpile	>=	0.055								
Stockpile	>=	0.060								
Stockpile	>=	0.065								
Stockpile	>=	0.070								
Stockpile	>=	0.075								
Stockpile	>=	0.080								

Source: *AMPL, November 21, 2025*

14.13 MINERAL RESOURCE ESTIMATES

The Resource was initially based solely on distance to drill hole or blast hole. The confidence level of some of the zones were then, in some cases, lowered for various reasons. As such, both calculations are shown as an aid in reconciling which areas may need more drilling to upgrade the various Resource confidence and which areas may need additional work of another nature (see Table 14.14 to Table 14.20, below).

TABLE 14.14
HXGN MINE PLAN™ 3D DENAK NORTHWEST LENS 1

Zone		Cut-off % MoS_2	Measured Resources Tonnes	Measured Resources % MoS_2		Indicated Resources Tonnes	Indicated Resources % MoS_2		Measured and Indicated Resources Tonnes	Measured and Indicated Resources % MoS_2		Inferred Resources Tonnes	Inferred Resources % MoS_2
Denak NorthWest	>=	0.010	92,311,000	0.048		38,270,000	0.041		130,581,000	0.046		3,559,000	0.031
Denak NorthWest	>=	0.015	79,014,000	0.054		34,018,000	0.045		113,032,000	0.051		2,982,000	0.034
Denak NorthWest	>=	0.020	68,787,000	0.059		29,663,000	0.049		98,449,000	0.056		2,478,000	0.038
Denak NorthWest	>=	0.025	58,983,000	0.066		25,023,000	0.054		84,006,000	0.062		1,969,000	0.042
Denak NorthWest	>=	0.030	51,264,000	0.071		21,056,000	0.059		72,320,000	0.068		1,465,000	0.047
Denak NorthWest	>=	0.035	43,570,000	0.078		17,328,000	0.065		60,898,000	0.075		1,107,000	0.051
Denak NorthWest	>=	0.040	37,435,000	0.085		14,282,000	0.071		51,717,000	0.081		821,000	0.056
Denak NorthWest	>=	0.045	32,215,000	0.092		11,408,000	0.079		43,623,000	0.089		598,000	0.062
Denak NorthWest	>=	0.050	27,726,000	0.100		9,360,000	0.086		37,086,000	0.096		473,000	0.066
Denak NorthWest	>=	0.055	23,892,000	0.107		7,624,000	0.093		31,516,000	0.104		354,000	0.070
Denak NorthWest	>=	0.060	20,834,000	0.115		6,342,000	0.101		27,176,000	0.111		258,000	0.075
Denak NorthWest	>=	0.065	18,158,000	0.122		5,377,000	0.108		23,534,000	0.119		200,000	0.079
Denak NorthWest	>=	0.070	15,981,000	0.130		4,614,000	0.114		20,595,000	0.126		152,000	0.083
Denak NorthWest	>=	0.075	14,073,000	0.138		3,980,000	0.121		18,053,000	0.134		101,000	0.088
Denak NorthWest	>=	0.080	12,388,000	0.146		3,557,000	0.126		15,945,000	0.141		69,000	0.093

Source: *AMPL, November 21, 2025*

TABLE 14.15
HXGN MINE PLAN™ 3D DENAK PYRITE LENS 2

Zone		Cut-off % MoS_2	Measured Resources Tonnes	Measured Resources % MoS_2	Indicated Resources Tonnes	Indicated Resources % MoS_2	Measured and Indicated Resources Tonnes	Measured and Indicated Resources % MoS_2	Inferred Resources Tonnes	Inferred Resources % MoS_2
Denak Pyrite Zone	>=	0.010	22,074,000	0.034	11,099,000	0.029	33,173,000	0.033	958,000	0.018
Denak Pyrite Zone	>=	0.015	14,268,000	0.047	8,039,000	0.036	22,306,000	0.043	490,000	0.025
Denak Pyrite Zone	>=	0.020	11,460,000	0.054	6,222,000	0.041	17,682,000	0.050	318,000	0.029
Denak Pyrite Zone	>=	0.025	9,232,000	0.062	4,649,000	0.048	13,881,000	0.057	169,000	0.034
Denak Pyrite Zone	>=	0.030	7,500,000	0.070	3,466,000	0.055	10,966,000	0.065	76,000	0.044
Denak Pyrite Zone	>=	0.035	6,119,000	0.079	2,560,000	0.063	8,679,000	0.074	43,000	0.053
Denak Pyrite Zone	>=	0.040	5,146,000	0.087	2,068,000	0.070	7,214,000	0.082	35,000	0.057
Denak Pyrite Zone	>=	0.045	4,331,000	0.095	1,679,000	0.076	6,010,000	0.090	29,000	0.060
Denak Pyrite Zone	>=	0.050	3,782,000	0.103	1,431,000	0.081	5,212,000	0.097	25,000	0.062
Denak Pyrite Zone	>=	0.055	3,368,000	0.109	1,229,000	0.086	4,597,000	0.103	21,000	0.064
Denak Pyrite Zone	>=	0.060	3,032,000	0.114	1,031,000	0.092	4,063,000	0.109	15,000	0.067
Denak Pyrite Zone	>=	0.065	2,645,000	0.122	894,000	0.096	3,539,000	0.116	12,000	0.068
Denak Pyrite Zone	>=	0.070	2,357,000	0.129	760,000	0.102	3,116,000	0.122	1,000	0.096
Denak Pyrite Zone	>=	0.075	2,145,000	0.134	654,000	0.106	2,799,000	0.128	-	0.112
Denak Pyrite Zone	>=	0.080	1,975,000	0.139	541,000	0.113	2,515,000	0.134	-	0.112

Source: *AMPL, November 21, 2025*

TABLE 14.16
HXGN MINE PLAN™ 3D DENAK SUB-EAST LENS 3

Zone		Cut-off	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
		% MoS_2	Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Denak Sub-East	>=	0.010	84,234,000	0.053		51,698,000	0.049		135,931,000	0.051		3,134,000	0.052
Denak Sub-East	>=	0.015	79,783,000	0.055		49,766,000	0.050		129,549,000	0.053		3,108,000	0.052
Denak Sub-East	>=	0.020	74,838,000	0.057		47,158,000	0.052		121,996,000	0.055		3,052,000	0.053
Denak Sub-East	>=	0.025	66,953,000	0.062		42,931,000	0.055		109,884,000	0.059		2,963,000	0.054
Denak Sub-East	>=	0.030	60,757,000	0.065		38,811,000	0.058		99,568,000	0.062		2,740,000	0.056
Denak Sub-East	>=	0.035	53,316,000	0.070		33,559,000	0.062		86,875,000	0.067		2,394,000	0.059
Denak Sub-East	>=	0.040	45,988,000	0.075		28,570,000	0.066		74,558,000	0.072		2,073,000	0.063
Denak Sub-East	>=	0.045	39,116,000	0.081		23,786,000	0.071		62,902,000	0.077		1,775,000	0.067
Denak Sub-East	>=	0.050	33,071,000	0.087		19,604,000	0.076		52,676,000	0.083		1,510,000	0.070
Denak Sub-East	>=	0.055	28,353,000	0.093		15,974,000	0.082		44,326,000	0.089		1,280,000	0.073
Denak Sub-East	>=	0.060	24,401,000	0.098		13,219,000	0.087		37,620,000	0.094		1,065,000	0.076
Denak Sub-East	>=	0.065	20,920,000	0.105		10,674,000	0.093		31,594,000	0.101		846,000	0.080
Denak Sub-East	>=	0.070	18,017,000	0.111		8,858,000	0.099		26,875,000	0.107		653,000	0.084
Denak Sub-East	>=	0.075	15,522,000	0.117		7,279,000	0.104		22,801,000	0.113		504,000	0.088
Denak Sub-East	>=	0.080	13,325,000	0.123		5,962,000	0.110		19,286,000	0.119		400,000	0.091

Source: *AMPL, November 21, 2025*

Table 14.17
HxGN Mine Plan™ 3D Denak V-S06 Lens 4

Zone		Cut-off	Measured Resources		Indicated Resources		Measured and Indicated Resources		Inferred Resources	
		% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2
Denak V-S06	>=	0.010	3,209,000	0.065	1,207,000	0.057	4,416,000	0.063	159,000	0.042
Denak V-S06	>=	0.015	2,874,000	0.071	1,127,000	0.060	4,001,000	0.068	159,000	0.042
Denak V-S06	>=	0.020	2,661,000	0.075	1,077,000	0.062	3,739,000	0.071	158,000	0.042
Denak V-S06	>=	0.025	2,390,000	0.082	996,000	0.065	3,386,000	0.077	154,000	0.043
Denak V-S06	>=	0.030	2,138,000	0.088	897,000	0.070	3,035,000	0.082	127,000	0.045
Denak V-S06	>=	0.035	1,893,000	0.095	820,000	0.073	2,713,000	0.089	89,000	0.048
Denak V-S06	>=	0.040	1,662,000	0.103	730,000	0.078	2,392,000	0.095	73,000	0.050
Denak V-S06	>=	0.045	1,497,000	0.110	676,000	0.080	2,173,000	0.101	32,000	0.054
Denak V-S06	>=	0.050	1,405,000	0.114	598,000	0.085	2,004,000	0.105	5,000	0.060
Denak V-S06	>=	0.055	1,319,000	0.118	544,000	0.088	1,863,000	0.109	1,000	0.080
Denak V-S06	>=	0.060	1,238,000	0.122	476,000	0.092	1,714,000	0.114	1,000	0.084
Denak V-S06	>=	0.065	1,101,000	0.130	396,000	0.099	1,497,000	0.122	1,000	0.084
Denak V-S06	>=	0.070	1,002,000	0.136	328,000	0.105	1,330,000	0.128	-	0.094
Denak V-S06	>=	0.075	918,000	0.142	288,000	0.110	1,205,000	0.134	-	0.138
Denak V-S06	>=	0.080	841,000	0.148	264,000	0.113	1,105,000	0.139	-	0.138

Source: *AMPL, November 21, 2025*

TABLE 14.18
HXGN MINE PLAN™ 3D DENAK WEST LENS 5

Zone		Cut-off	Measured Resources		Indicated Resources		Measured and Indicated Resources		Inferred Resources	
		% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2
Denak West	>=	0.010	24,507,000	0.037	10,619,000	0.035	35,127,000	0.036	345,000	0.034
Denak West	>=	0.015	20,576,000	0.042	9,195,000	0.039	29,771,000	0.041	307,000	0.035
Denak West	>=	0.020	17,545,000	0.046	7,665,000	0.043	25,210,000	0.045	211,000	0.042
Denak West	>=	0.025	13,891,000	0.052	5,954,000	0.049	19,845,000	0.051	166,000	0.046
Denak West	>=	0.030	11,265,000	0.058	4,469,000	0.056	15,734,000	0.057	133,000	0.049
Denak West	>=	0.035	8,583,000	0.066	3,491,000	0.063	12,074,000	0.065	109,000	0.052
Denak West	>=	0.040	7,009,000	0.073	2,754,000	0.070	9,763,000	0.072	90,000	0.054
Denak West	>=	0.045	5,803,000	0.079	2,182,000	0.077	7,985,000	0.079	50,000	0.059
Denak West	>=	0.050	4,791,000	0.086	1,783,000	0.084	6,574,000	0.085	12,000	0.083
Denak West	>=	0.055	3,811,000	0.095	1,457,000	0.091	5,268,000	0.094	6,000	0.110
Denak West	>=	0.060	3,136,000	0.103	1,245,000	0.097	4,380,000	0.101	6,000	0.114
Denak West	>=	0.065	2,663,000	0.110	1,037,000	0.104	3,700,000	0.108	6,000	0.114
Denak West	>=	0.070	2,350,000	0.116	875,000	0.111	3,225,000	0.115	6,000	0.114
Denak West	>=	0.075	2,058,000	0.122	761,000	0.117	2,819,000	0.121	6,000	0.114
Denak West	>=	0.080	1,792,000	0.129	665,000	0.123	2,457,000	0.127	-	

Source: *AMPL, November 21, 2025*

TABLE 14.19
DENAK WEST FAULT LENS 6

Zone		Cut-off	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
		% MoS_2	Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Denak West Fault	>=	0.010	11,078,000	0.037		4,467,000	0.035		15,545,000	0.036		31,000	0.039
Denak West Fault	>=	0.015	9,668,000	0.041		4,127,000	0.036		13,795,000	0.040		31,000	0.039
Denak West Fault	>=	0.020	8,351,000	0.045		3,612,000	0.039		11,963,000	0.043		31,000	0.039
Denak West Fault	>=	0.025	6,513,000	0.051		3,028,000	0.042		9,542,000	0.048		30,000	0.039
Denak West Fault	>=	0.030	5,365,000	0.056		2,502,000	0.046		7,867,000	0.053		30,000	0.039
Denak West Fault	>=	0.035	4,112,000	0.064		1,827,000	0.051		5,938,000	0.060		24,000	0.041
Denak West Fault	>=	0.040	3,433,000	0.069		1,348,000	0.056		4,781,000	0.065		16,000	0.043
Denak West Fault	>=	0.045	2,761,000	0.076		897,000	0.063		3,659,000	0.073		1,000	0.062
Denak West Fault	>=	0.050	2,346,000	0.081		677,000	0.068		3,023,000	0.078		1,000	0.072
Denak West Fault	>=	0.055	1,919,000	0.088		485,000	0.075		2,403,000	0.085		1,000	0.084
Denak West Fault	>=	0.060	1,605,000	0.094		379,000	0.080		1,984,000	0.091		1,000	0.086
Denak West Fault	>=	0.065	1,384,000	0.099		290,000	0.085		1,673,000	0.096		-	0.087
Denak West Fault	>=	0.070	1,229,000	0.103		223,000	0.090		1,452,000	0.101		-	0.091
Denak West Fault	>=	0.075	1,060,000	0.108		174,000	0.096		1,234,000	0.106		-	0.091
Denak West Fault	>=	0.080	948,000	0.111		143,000	0.100		1,090,000	0.110		-	0.091

Source: *AMPL, November 21, 2025*

Table 14.20 HxGN Mine Plant™ 3D Endako – E-Kspar Lens 7 – as Calculated Source – Category Confidence Subsquently Reduced													
Zone		Cut-off	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
		% MoS_2	Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Endako K-Spar	>=	0.010	26,067,000	0.037		26,640,000	0.037		52,707,000	0.037		19,120,000	0.039
Endako K-Spar	>=	0.015	23,966,000	0.039		25,591,000	0.038		49,557,000	0.038		18,927,000	0.039
Endako K-Spar	>=	0.020	22,179,000	0.041		24,101,000	0.039		46,280,000	0.040		18,574,000	0.040
Endako K-Spar	>=	0.025	19,452,000	0.044		21,709,000	0.041		41,161,000	0.042		17,634,000	0.041
Endako K-Spar	>=	0.030	15,531,000	0.048		17,609,000	0.044		33,141,000	0.046		15,947,000	0.042
Endako K-Spar	>=	0.035	12,144,000	0.052		13,233,000	0.048		25,377,000	0.050		13,842,000	0.044
Endako K-Spar	>=	0.040	9,129,000	0.057		9,833,000	0.052		18,962,000	0.055		9,642,000	0.047
Endako K-Spar	>=	0.045	6,605,000	0.063		6,668,000	0.057		13,273,000	0.060		5,313,000	0.050
Endako K-Spar	>=	0.050	5,068,000	0.068		4,600,000	0.061		9,668,000	0.065		1,931,000	0.056
Endako K-Spar	>=	0.055	3,654,000	0.075		2,703,000	0.068		6,357,000	0.072		939,000	0.061
Endako K-Spar	>=	0.060	2,603,000	0.082		1,828,000	0.073		4,431,000	0.078		449,000	0.066
Endako K-Spar	>=	0.065	1,960,000	0.089		1,266,000	0.078		3,226,000	0.085		208,000	0.072
Endako K-Spar	>=	0.070	1,519,000	0.095		926,000	0.083		2,446,000	0.090		104,000	0.076
Endako K-Spar	>=	0.075	1,191,000	0.101		648,000	0.087		1,839,000	0.096		49,000	0.081
Endako K-Spar	>=	0.080	961,000	0.107		528,000	0.090		1,489,000	0.101		21,000	0.087

Source: *AMPL, November 21, 2025*

The Endako K-Spar Zone subsequently had its confidence level reduced due to some uncertainty of the state of the pit walls, which experienced some sloughing (see Table 14.21 to Table 14.22, below).

TABLE 14.21
HXGN MINE PLAN™ 3D ENDAKO – E-KSPAR LENS 7 – AS REPORTED

Zone		Cut-off	Measured Resources		Indicated Resources		Measured and Indicated Resources		Inferred Resources	
		% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2
Endako K-Spar	>=	0.010			52,707,000	0.037	52,707,000	0.037	19,120,000	0.039
Endako K-Spar	>=	0.015			49,557,000	0.038	49,557,000	0.038	18,927,000	0.039
Endako K-Spar	>=	0.020			46,280,000	0.040	46,280,000	0.040	18,574,000	0.040
Endako K-Spar	>=	0.025			41,161,000	0.042	41,161,000	0.042	17,634,000	0.041
Endako K-Spar	>=	0.030			33,141,000	0.046	33,141,000	0.046	15,947,000	0.042
Endako K-Spar	>=	0.035			25,377,000	0.050	25,377,000	0.050	13,842,000	0.044
Endako K-Spar	>=	0.040			18,962,000	0.055	18,962,000	0.055	9,642,000	0.047
Endako K-Spar	>=	0.045			13,273,000	0.060	13,273,000	0.060	5,313,000	0.050
Endako K-Spar	>=	0.050			9,668,000	0.065	9,668,000	0.065	1,931,000	0.056
Endako K-Spar	>=	0.055			6,357,000	0.072	6,357,000	0.072	939,000	0.061
Endako K-Spar	>=	0.060			4,431,000	0.078	4,431,000	0.078	449,000	0.066
Endako K-Spar	>=	0.065			3,226,000	0.085	3,226,000	0.085	208,000	0.072
Endako K-Spar	>=	0.070			2,446,000	0.090	2,446,000	0.090	104,000	0.076
Endako K-Spar	>=	0.075			1,839,000	0.096	1,839,000	0.096	49,000	0.081
Endako K-Spar	>=	0.080			1,489,000	0.101	1,489,000	0.101	21,000	0.087

Source: *AMPL, November 21, 2025*

TABLE 14.22
HXGN MINE PLAN™ 3D ENDAKO – PYRITE – LENS 9 AS CALCULATED
SOURCE – CATEGORY CONFIDENCE SUBSEQUENTLY REDUCED

Zone		Cut-off	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
		% MoS_2	Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Endako Pyrite	>=	0.010	40,312,000	0.040		28,739,000	0.035		69,051,000	0.038		23,061,000	0.024
Endako Pyrite	>=	0.015	35,823,000	0.044		24,850,000	0.039		60,673,000	0.042		16,376,000	0.028
Endako Pyrite	>=	0.020	31,932,000	0.047		20,886,000	0.043		52,818,000	0.046		10,957,000	0.034
Endako Pyrite	>=	0.025	27,708,000	0.051		17,416,000	0.047		45,125,000	0.050		8,048,000	0.038
Endako Pyrite	>=	0.030	24,259,000	0.055		14,779,000	0.051		39,039,000	0.053		6,334,000	0.042
Endako Pyrite	>=	0.035	20,205,000	0.059		11,925,000	0.055		32,130,000	0.058		4,694,000	0.045
Endako Pyrite	>=	0.040	16,757,000	0.064		9,531,000	0.060		26,288,000	0.062		3,415,000	0.048
Endako Pyrite	>=	0.045	13,604,000	0.069		7,517,000	0.065		21,121,000	0.067		2,104,000	0.051
Endako Pyrite	>=	0.050	11,294,000	0.073		6,158,000	0.069		17,452,000	0.072		992,000	0.057
Endako Pyrite	>=	0.055	9,270,000	0.078		4,874,000	0.073		14,144,000	0.076		370,000	0.065
Endako Pyrite	>=	0.060	7,739,000	0.082		3,860,000	0.077		11,599,000	0.080		232,000	0.070
Endako Pyrite	>=	0.065	6,240,000	0.087		3,015,000	0.082		9,256,000	0.085		152,000	0.074
Endako Pyrite	>=	0.070	5,098,000	0.091		2,335,000	0.086		7,433,000	0.090		99,000	0.078
Endako Pyrite	>=	0.075	4,038,000	0.096		1,711,000	0.091		5,748,000	0.095		54,000	0.084
Endako Pyrite	>=	0.080	3,164,000	0.102		1,254,000	0.097		4,417,000	0.100		33,000	0.088

Source: *AMPL, November 21, 2025*

The Endako Pyrite subsequently had its confidence level reduced due to some uncertainty of the state of the pit walls, which experienced some sloughing. In addtion, it may require some additional metallurgical work to improve recoverablity (see Table 14.23 to Table 14.24, below).

TABLE 14.23
HXGN MINE PLAN™ 3D ENDAKO – PYRITE – LENS 9 – AS REPORTED

Zone		Cut-off	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
		% MoS_2	Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Endako Pyrite	>=	0.010				69,051,000	0.038		69,051,000	0.038		23,061,000	0.024
Endako Pyrite	>=	0.015				60,673,000	0.042		60,673,000	0.042		16,376,000	0.028
Endako Pyrite	>=	0.020				52,818,000	0.046		52,818,000	0.046		10,957,000	0.034
Endako Pyrite	>=	0.025				45,125,000	0.050		45,125,000	0.050		8,048,000	0.038
Endako Pyrite	>=	0.030				39,039,000	0.053		39,039,000	0.053		6,334,000	0.042
Endako Pyrite	>=	0.035				32,130,000	0.058		32,130,000	0.058		4,694,000	0.045
Endako Pyrite	>=	0.040				26,288,000	0.062		26,288,000	0.062		3,415,000	0.048
Endako Pyrite	>=	0.045				21,121,000	0.067		21,121,000	0.067		2,104,000	0.051
Endako Pyrite	>=	0.050				17,452,000	0.072		17,452,000	0.072		992,000	0.057
Endako Pyrite	>=	0.055				14,144,000	0.076		14,144,000	0.076		370,000	0.065
Endako Pyrite	>=	0.060				11,599,000	0.080		11,599,000	0.080		232,000	0.070
Endako Pyrite	>=	0.065				9,256,000	0.085		9,256,000	0.085		152,000	0.074
Endako Pyrite	>=	0.070				7,433,000	0.090		7,433,000	0.090		99,000	0.078
Endako Pyrite	>=	0.075				5,748,000	0.095		5,748,000	0.095		54,000	0.084
Endako Pyrite	>=	0.080				4,417,000	0.100		4,417,000	0.100		33,000	0.088

Source: *AMPL, November 21, 2025*

TABLE 14.24
HXGN MINE PLAN™ 3D ENDAKO – SOUTH BASAL – LENS 10 – AS CALCULATED
SOURCE – CATEGORY CONFIDENCE – SUBSEQUENTLY REDUCED

Zone		Cut-off	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
		% MoS_2	Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Endako S-Basal	>=	0.010	8,684,000	0.074		5,112,000	0.071		13,796,000	0.073		1,699,000	0.053
Endako S-Basal	>=	0.015	8,367,000	0.077		4,915,000	0.073		13,281,000	0.075		1,549,000	0.057
Endako S-Basal	>=	0.020	8,061,000	0.079		4,649,000	0.076		12,710,000	0.078		1,508,000	0.059
Endako S-Basal	>=	0.025	7,689,000	0.082		4,367,000	0.080		12,056,000	0.081		1,460,000	0.060
Endako S-Basal	>=	0.030	7,145,000	0.086		4,072,000	0.083		11,217,000	0.085		1,277,000	0.064
Endako S-Basal	>=	0.035	6,668,000	0.090		3,763,000	0.088		10,431,000	0.089		1,044,000	0.072
Endako S-Basal	>=	0.040	6,197,000	0.094		3,483,000	0.092		9,680,000	0.093		812,000	0.082
Endako S-Basal	>=	0.045	5,540,000	0.100		3,059,000	0.099		8,599,000	0.100		689,000	0.089
Endako S-Basal	>=	0.050	4,984,000	0.106		2,759,000	0.104		7,743,000	0.105		645,000	0.092
Endako S-Basal	>=	0.055	4,478,000	0.112		2,390,000	0.112		6,868,000	0.112		585,000	0.096
Endako S-Basal	>=	0.060	4,132,000	0.117		2,131,000	0.119		6,263,000	0.118		510,000	0.101
Endako S-Basal	>=	0.065	3,690,000	0.123		1,888,000	0.127		5,578,000	0.124		473,000	0.104
Endako S-Basal	>=	0.070	3,305,000	0.130		1,630,000	0.136		4,935,000	0.132		440,000	0.107
Endako S-Basal	>=	0.075	2,897,000	0.138		1,471,000	0.143		4,367,000	0.140		381,000	0.113
Endako S-Basal	>=	0.080	2,548,000	0.146		1,338,000	0.150		3,886,000	0.147		348,000	0.116

Source: *AMPL, November 21, 2025*

The Endako S-Basal subsequently had its confidence level reduced due to some uncertainty of the state of the pit walls, which experienced some sloughing (see Table 14.25 to Table 14.26, below).

TABLE 14.25
HXGN MINE PLAN™ 3D ENDAKO – SOUTH BASAL – LENS 10 – AS REPORTED

Zone		Cut-off	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
		% MoS_2	Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Endako S-Basal	>=	0.010				13,796,000	0.073		13,796,000	0.073		1,699,000	0.053
Endako S-Basal	>=	0.015				13,281,000	0.075		13,281,000	0.075		1,549,000	0.057
Endako S-Basal	>=	0.020				12,710,000	0.078		12,710,000	0.078		1,508,000	0.059
Endako S-Basal	>=	0.025				12,056,000	0.081		12,056,000	0.081		1,460,000	0.060
Endako S-Basal	>=	0.030				11,217,000	0.085		11,217,000	0.085		1,277,000	0.064
Endako S-Basal	>=	0.035				10,431,000	0.089		10,431,000	0.089		1,044,000	0.072
Endako S-Basal	>=	0.040				9,680,000	0.093		9,680,000	0.093		812,000	0.082
Endako S-Basal	>=	0.045				8,599,000	0.100		8,599,000	0.100		689,000	0.089
Endako S-Basal	>=	0.050				7,743,000	0.105		7,743,000	0.105		645,000	0.092
Endako S-Basal	>=	0.055				6,868,000	0.112		6,868,000	0.112		585,000	0.096
Endako S-Basal	>=	0.060				6,263,000	0.118		6,263,000	0.118		510,000	0.101
Endako S-Basal	>=	0.065				5,578,000	0.124		5,578,000	0.124		473,000	0.104
Endako S-Basal	>=	0.070				4,935,000	0.132		4,935,000	0.132		440,000	0.107
Endako S-Basal	>=	0.075				4,367,000	0.140		4,367,000	0.140		381,000	0.113
Endako S-Basal	>=	0.080				3,886,000	0.147		3,886,000	0.147		348,000	0.116

Source: *AMPL, November 21, 2025*

TABLE 14.26
HXGN MINE PLAN™ 3D ENDAKO – SUB EAST EXTENSION – LENS 11 – AS CALCULATED SOURCE CATEGORY CONFIDENCE SUBSEQUENTLY REDUCED

Zone		Cut-off % MoS_2	Measured Resources		Indicated Resources		Measured and Indicated Resources		Inferred Resources	
			Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2
Endakio Sub E -extn	>=	0.010	58,993,000	0.060	47,152,000	0.054	106,145,000	0.057	9,079,000	0.039
Endakio Sub E -extn	>=	0.015	58,307,000	0.061	46,427,000	0.055	104,734,000	0.058	8,742,000	0.041
Endakio Sub E -extn	>=	0.020	57,060,000	0.062	45,241,000	0.056	102,302,000	0.059	8,329,000	0.042
Endakio Sub E -extn	>=	0.025	53,888,000	0.064	42,983,000	0.058	96,871,000	0.061	7,543,000	0.044
Endakio Sub E -extn	>=	0.030	50,723,000	0.066	40,269,000	0.060	90,992,000	0.063	6,375,000	0.047
Endakio Sub E -extn	>=	0.035	46,075,000	0.070	36,724,000	0.062	82,799,000	0.066	5,204,000	0.050
Endakio Sub E -extn	>=	0.040	41,299,000	0.073	32,747,000	0.066	74,047,000	0.070	4,044,000	0.054
Endakio Sub E -extn	>=	0.045	36,322,000	0.078	28,333,000	0.069	64,655,000	0.074	2,946,000	0.058
Endakio Sub E -extn	>=	0.050	32,039,000	0.082	24,210,000	0.073	56,249,000	0.078	2,250,000	0.061
Endakio Sub E -extn	>=	0.055	27,750,000	0.086	20,068,000	0.077	47,818,000	0.083	1,522,000	0.066
Endakio Sub E -extn	>=	0.060	24,446,000	0.090	16,741,000	0.081	41,187,000	0.087	974,000	0.071
Endakio Sub E -extn	>=	0.065	20,702,000	0.095	13,418,000	0.086	34,120,000	0.092	628,000	0.075
Endakio Sub E -extn	>=	0.070	17,782,000	0.100	10,881,000	0.091	28,663,000	0.097	441,000	0.079
Endakio Sub E -extn	>=	0.075	14,758,000	0.106	8,267,000	0.097	23,026,000	0.103	210,000	0.087
Endakio Sub E -extn	>=	0.080	12,219,000	0.112	6,575,000	0.102	18,794,000	0.108	158,000	0.090

Source: *AMPL, November 21, 2025*

The Endako Sub East Extension subsequently had its confidence level reduced due to some uncertainty of the state of the pit walls, which experienced some sloughing (see Table 14.27 to Table 14.28, below).

TABLE 14.27
HXGN MINE PLAN™ 3D ENDAKO – SUB EAST EXTENSION – LENS 11 – AS REPORTED

Zone		Cut-off % MoS_2	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
			Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Endakio Sub E -extn	>=	0.010				106,145,000	0.057		106,145,000	0.057		9,079,000	0.039
Endakio Sub E -extn	>=	0.015				104,734,000	0.058		104,734,000	0.058		8,742,000	0.041
Endakio Sub E -extn	>=	0.020				102,302,000	0.059		102,302,000	0.059		8,329,000	0.042
Endakio Sub E -extn	>=	0.025				96,871,000	0.061		96,871,000	0.061		7,543,000	0.044
Endakio Sub E -extn	>=	0.030				90,992,000	0.063		90,992,000	0.063		6,375,000	0.047
Endakio Sub E -extn	>=	0.035				82,799,000	0.066		82,799,000	0.066		5,204,000	0.050
Endakio Sub E -extn	>=	0.040				74,047,000	0.070		74,047,000	0.070		4,044,000	0.054
Endakio Sub E -extn	>=	0.045				64,655,000	0.074		64,655,000	0.074		2,946,000	0.058
Endakio Sub E -extn	>=	0.050				56,249,000	0.078		56,249,000	0.078		2,250,000	0.061
Endakio Sub E -extn	>=	0.055				47,818,000	0.083		47,818,000	0.083		1,522,000	0.066
Endakio Sub E -extn	>=	0.060				41,187,000	0.087		41,187,000	0.087		974,000	0.071
Endakio Sub E -extn	>=	0.065				34,120,000	0.092		34,120,000	0.092		628,000	0.075
Endakio Sub E -extn	>=	0.070				28,663,000	0.097		28,663,000	0.097		441,000	0.079
Endakio Sub E -extn	>=	0.075				23,026,000	0.103		23,026,000	0.103		210,000	0.087
Endakio Sub E -extn	>=	0.080				18,794,000	0.108		18,794,000	0.108		158,000	0.090

Source: *AMPL, November 21, 2025*

TABLE 14.28
HXGN MINE PLAN™ 3D ENDAKO – SUB WEST EXTENSION – LENS 12 – AS CALCULATED SOURCE – CATEGORY CONFIDENCE SUBSQUENTLY REDUCED

Zone		Cut-off % MoS_2	Measured Resources		Indicated Resources		Measured and Indicated Resources		Inferred Resources	
			Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2
Endako Sub W	>=	0.010	38,245,000	0.058	24,702,000	0.057	62,947,000	0.057	3,175,000	0.040
Endako Sub W	>=	0.015	37,120,000	0.059	24,380,000	0.058	61,500,000	0.058	3,128,000	0.040
Endako Sub W	>=	0.020	35,783,000	0.060	23,782,000	0.059	59,565,000	0.060	2,905,000	0.042
Endako Sub W	>=	0.025	33,428,000	0.063	22,707,000	0.061	56,135,000	0.062	2,650,000	0.044
Endako Sub W	>=	0.030	31,079,000	0.066	21,466,000	0.063	52,545,000	0.064	2,204,000	0.047
Endako Sub W	>=	0.035	28,207,000	0.069	19,544,000	0.066	47,751,000	0.068	1,814,000	0.051
Endako Sub W	>=	0.040	25,285,000	0.073	17,408,000	0.069	42,693,000	0.071	1,477,000	0.054
Endako Sub W	>=	0.045	21,683,000	0.078	14,993,000	0.073	36,677,000	0.076	1,158,000	0.057
Endako Sub W	>=	0.050	18,757,000	0.083	12,949,000	0.078	31,705,000	0.081	816,000	0.062
Endako Sub W	>=	0.055	16,464,000	0.087	10,863,000	0.082	27,327,000	0.085	553,000	0.066
Endako Sub W	>=	0.060	14,267,000	0.092	9,270,000	0.087	23,537,000	0.090	370,000	0.071
Endako Sub W	>=	0.065	12,121,000	0.097	7,653,000	0.092	19,774,000	0.095	209,000	0.078
Endako Sub W	>=	0.070	10,494,000	0.102	6,412,000	0.097	16,905,000	0.100	129,000	0.084
Endako Sub W	>=	0.075	8,973,000	0.107	5,366,000	0.102	14,339,000	0.105	88,000	0.090
Endako Sub W	>=	0.080	7,628,000	0.113	4,544,000	0.106	12,172,000	0.110	72,000	0.093

Source: *AMPL, November 21, 2025*

The Endako Sub West Extension subsequently had its confidence level reduced due to some uncertainty of the state of the pit walls, which experienced some sloughing (see Table 14.29 to Table 14.30, below).

Table 14.29
HxGN Mine Plan™ 3D Endako – Sub West Extension – Lens 12 – as Reported

Zone		Cut-off % MoS_2	Measured Resources		Indicated Resources		Measured and Indicated Resources		Inferred Resources	
			Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2
Endako Sub W	>=	0.010			62,947,000	0.057	62,947,000	0.057	3,175,000	0.040
Endako Sub W	>=	0.015			61,500,000	0.058	61,500,000	0.058	3,128,000	0.040
Endako Sub W	>=	0.020			59,565,000	0.060	59,565,000	0.060	2,905,000	0.042
Endako Sub W	>=	0.025			56,135,000	0.062	56,135,000	0.062	2,650,000	0.044
Endako Sub W	>=	0.030			52,545,000	0.064	52,545,000	0.064	2,204,000	0.047
Endako Sub W	>=	0.035			47,751,000	0.068	47,751,000	0.068	1,814,000	0.051
Endako Sub W	>=	0.040			42,693,000	0.071	42,693,000	0.071	1,477,000	0.054
Endako Sub W	>=	0.045			36,677,000	0.076	36,677,000	0.076	1,158,000	0.057
Endako Sub W	>=	0.050			31,705,000	0.081	31,705,000	0.081	816,000	0.062
Endako Sub W	>=	0.055			27,327,000	0.085	27,327,000	0.085	553,000	0.066
Endako Sub W	>=	0.060			23,537,000	0.090	23,537,000	0.090	370,000	0.071
Endako Sub W	>=	0.065			19,774,000	0.095	19,774,000	0.095	209,000	0.078
Endako Sub W	>=	0.070			16,905,000	0.100	16,905,000	0.100	129,000	0.084
Endako Sub W	>=	0.075			14,339,000	0.105	14,339,000	0.105	88,000	0.090
Endako Sub W	>=	0.080			12,172,000	0.110	12,172,000	0.110	72,000	0.093

***Source:** AMPL, November 21, 2025*

TABLE 14.30
HXGN MINE PLAN™ 3D ENDAKO – VEIN – LENS 13 – AS CALCULATED SOURCE – CATEGORY CONFIDENCE SUBSEQUENTLY REDUCED

Zone		Cut-off % MoS_2	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
			Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Endako Vein		0.010	1,722,000	0.109		1,291,000	0.114		3,013,000	0.111		441,000	0.099
Endako Vein		0.015	1,722,000	0.109		1,291,000	0.114		3,013,000	0.111		441,000	0.099
Endako Vein	>=	0.020	1,722,000	0.109		1,291,000	0.114		3,013,000	0.111		441,000	0.099
Endako Vein	>=	0.025	1,722,000	0.109		1,291,000	0.114		3,013,000	0.111		441,000	0.099
Endako Vein	>=	0.030	1,719,000	0.109		1,291,000	0.114		3,010,000	0.111		441,000	0.099
Endako Vein	>=	0.035	1,716,000	0.109		1,285,000	0.114		3,001,000	0.111		441,000	0.099
Endako Vein	>=	0.040	1,660,000	0.111		1,277,000	0.115		2,937,000	0.113		441,000	0.099
Endako Vein	>=	0.045	1,612,000	0.113		1,261,000	0.116		2,873,000	0.114		441,000	0.099
Endako Vein	>=	0.050	1,498,000	0.118		1,217,000	0.118		2,715,000	0.118		406,000	0.103
Endako Vein	>=	0.055	1,424,000	0.122		1,185,000	0.120		2,609,000	0.121		392,000	0.105
Endako Vein	>=	0.060	1,407,000	0.123		1,148,000	0.122		2,555,000	0.122		391,000	0.105
Endako Vein	>=	0.065	1,376,000	0.124		1,093,000	0.125		2,469,000	0.124		391,000	0.105
Endako Vein	>=	0.070	1,319,000	0.126		995,000	0.131		2,314,000	0.128		391,000	0.105
Endako Vein	>=	0.075	1,286,000	0.128		939,000	0.134		2,225,000	0.131		388,000	0.105
Endako Vein	>=	0.080	1,221,000	0.131		893,000	0.137		2,114,000	0.133		340,000	0.109

Source: *AMPL, November 21, 2025*

The Endako Vein subsequently had its confidence level reduced due to some uncertainty of the state of the pit walls, which experienced some sloughing (see Table 14.31 to Table 14.32, below).

Endako Mine Restart PEA (NI 43-101)
Effective Date: November 21, 2025; Filing Date: January 5, 2026

TABLE 14.31
HXGN MINE PLAN™ 3D ENDAKO – VEIN – LENS 13 – AS REPORTED

Zone		Cut-off % MoS_2	Measured Resources		Indicated Resources		Measured and Indicated Resources		Inferred Resources	
			Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	% MoS_2
Endako Vein		0.010			3,013,000	0.111	3,013,000	0.111	441,000	0.099
Endako Vein		0.015			3,013,000	0.111	3,013,000	0.111	441,000	0.099
Endako Vein	>=	0.020			3,013,000	0.111	3,013,000	0.111	441,000	0.099
Endako Vein	>=	0.025			3,013,000	0.111	3,013,000	0.111	441,000	0.099
Endako Vein	>=	0.030			3,010,000	0.111	3,010,000	0.111	441,000	0.099
Endako Vein	>=	0.035			3,001,000	0.111	3,001,000	0.111	441,000	0.099
Endako Vein	>=	0.040			2,937,000	0.113	2,937,000	0.113	441,000	0.099
Endako Vein	>=	0.045			2,873,000	0.114	2,873,000	0.114	441,000	0.099
Endako Vein	>=	0.050			2,715,000	0.118	2,715,000	0.118	406,000	0.103
Endako Vein	>=	0.055			2,609,000	0.121	2,609,000	0.121	392,000	0.105
Endako Vein	>=	0.060			2,555,000	0.122	2,555,000	0.122	391,000	0.105
Endako Vein	>=	0.065			2,469,000	0.124	2,469,000	0.124	391,000	0.105
Endako Vein	>=	0.070			2,314,000	0.128	2,314,000	0.128	391,000	0.105
Endako Vein	>=	0.075			2,225,000	0.131	2,225,000	0.131	388,000	0.105
Endako Vein	>=	0.080			2,114,000	0.133	2,114,000	0.133	340,000	0.109

Source: *AMPL, November 21, 2025*

TABLE 14.32
HXGN MINE PLAN™ 3D CASEY – LENS 14 AS CALCULATED SOURCE – CATEGORY CONFIDENCE SUBSEQUENTLY REDUCED

Zone		Cut-off % MoS_2	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
			Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Casey	>=	0.010	33,647,000	0.051		27,758,000	0.042		61,405,000	0.047		38,398,000	0.032
Casey	>=	0.015	31,422,000	0.054		24,226,000	0.046		55,648,000	0.051		32,203,000	0.036
Casey	>=	0.020	29,193,000	0.057		22,465,000	0.049		51,658,000	0.053		28,069,000	0.039
Casey	>=	0.025	26,652,000	0.060		19,693,000	0.052		46,345,000	0.057		22,931,000	0.042
Casey	>=	0.030	23,505,000	0.065		16,276,000	0.058		39,781,000	0.062		16,941,000	0.048
Casey	>=	0.035	20,036,000	0.070		13,250,000	0.063		33,286,000	0.068		12,837,000	0.053
Casey	>=	0.040	17,014,000	0.076		10,721,000	0.070		27,735,000	0.074		9,453,000	0.059
Casey	>=	0.045	14,560,000	0.082		8,768,000	0.076		23,328,000	0.080		7,306,000	0.064
Casey	>=	0.050	12,519,000	0.088		7,458,000	0.081		19,977,000	0.085		5,918,000	0.067
Casey	>=	0.055	10,756,000	0.094		6,335,000	0.086		17,091,000	0.091		4,638,000	0.072
Casey	>=	0.060	9,434,000	0.099		5,506,000	0.091		14,940,000	0.096		3,652,000	0.076
Casey	>=	0.065	8,381,000	0.103		4,623,000	0.096		13,004,000	0.101		2,773,000	0.080
Casey	>=	0.070	7,405,000	0.108		3,938,000	0.101		11,342,000	0.106		2,117,000	0.084
Casey	>=	0.075	6,467,000	0.113		3,414,000	0.106		9,881,000	0.111		1,614,000	0.088
Casey	>=	0.080	5,502,000	0.120		2,986,000	0.110		8,488,000	0.116		1,113,000	0.093

Source: *AMPL, November 21, 2025*

The Casey subsequently had its confidence level reduced due to the lack of a detailed pit design relative to the proximity of the tailings dam. In addition, the boundaries are not clearly defined and the zone requires more diamond drilling (see Table 14.33, below).

Table 14.33 HxGN Mine Plan™ 3D Casey – Lens 14 – as Reported													
Zone		Cut-off % MoS_2	Measured Resource			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
			Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Casey	>=	0.010										99,803,000	0.041
Casey	>=	0.015										87,851,000	0.045
Casey	>=	0.020										79,727,000	0.048
Casey	>=	0.025										69,276,000	0.052
Casey	>=	0.030										56,722,000	0.058
Casey	>=	0.035										46,123,000	0.063
Casey	>=	0.040										37,188,000	0.070
Casey	>=	0.045										30,634,000	0.076
Casey	>=	0.050										25,895,000	0.081
Casey	>=	0.055										21,729,000	0.087
Casey	>=	0.060										18,592,000	0.092
Casey	>=	0.065										15,777,000	0.097
Casey	>=	0.070										13,459,000	0.102
Casey	>=	0.075										11,495,000	0.107
Casey	>=	0.080										9,601,000	0.113

Source: *AMPL, November 21, 2025*

14.14 RESULTS

Table 14.34, below, summarises the total Resources tabulated for the property including the existing stockpiles and all the various pit lenses. Figure 14.57, below shows the non-orthogonal view of the Resource by cut-off grade. Figure 14.58, below shows the east view of the Resource by cut-off grade in Section 26400.

TABLE 14.34
HXGN MINE PLAN™ 3D TOTAL RESOURCE

Zone		Cut-off % MoS_2	Measured Resources			Indicated Resources			Measured and Indicated Resources			Inferred Resources	
			Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2		Tonnes	% MoS_2
Total	>=	0.010	237,413,000	0.047		435,641,000	0.049		673,054,000	0.048		164,564,000	0.038
Total	>=	0.015	206,183,000	0.052		409,652,000	0.052		615,834,000	0.052		144,091,000	0.037
Total	>=	0.020	183,642,000	0.056		382,707,000	0.054		566,349,000	0.055		128,689,000	0.040
Total	>=	0.025	157,962,000	0.062		347,564,000	0.057		505,527,000	0.059		112,503,000	0.043
Total	>=	0.030	138,289,000	0.067		311,767,000	0.061		450,056,000	0.063		93,871,000	0.046
Total	>=	0.035	117,593,000	0.073		271,696,000	0.065		389,288,000	0.067		76,928,000	0.050
Total	>=	0.040	100,673,000	0.079		234,981,000	0.069		335,654,000	0.072		60,127,000	0.054
Total	>=	0.045	85,723,000	0.086		187,826,000	0.074		273,550,000	0.078		45,770,000	0.060
Total	>=	0.050	73,121,000	0.093		158,985,000	0.079		232,107,000	0.084		34,961,000	0.066
Total	>=	0.055	62,662,000	0.100		132,436,000	0.085		195,096,000	0.090		27,753,000	0.072
Total	>=	0.060	54,246,000	0.106		112,264,000	0.090		166,509,000	0.095		22,864,000	0.077
Total	>=	0.065	46,871,000	0.113		93,091,000	0.096		139,960,000	0.102		18,903,000	0.083
Total	>=	0.070	40,936,000	0.120		78,354,000	0.101		119,289,000	0.108		15,875,000	0.087
Total	>=	0.075	35,776,000	0.127		64,680,000	0.107		100,455,000	0.114		13,276,000	0.092
Total	>=	0.080	31,269,000	0.134		54,004,000	0.113		85,270,000	0.121		11,042,000	0.097

Source: *AMPL, November 21, 2025*

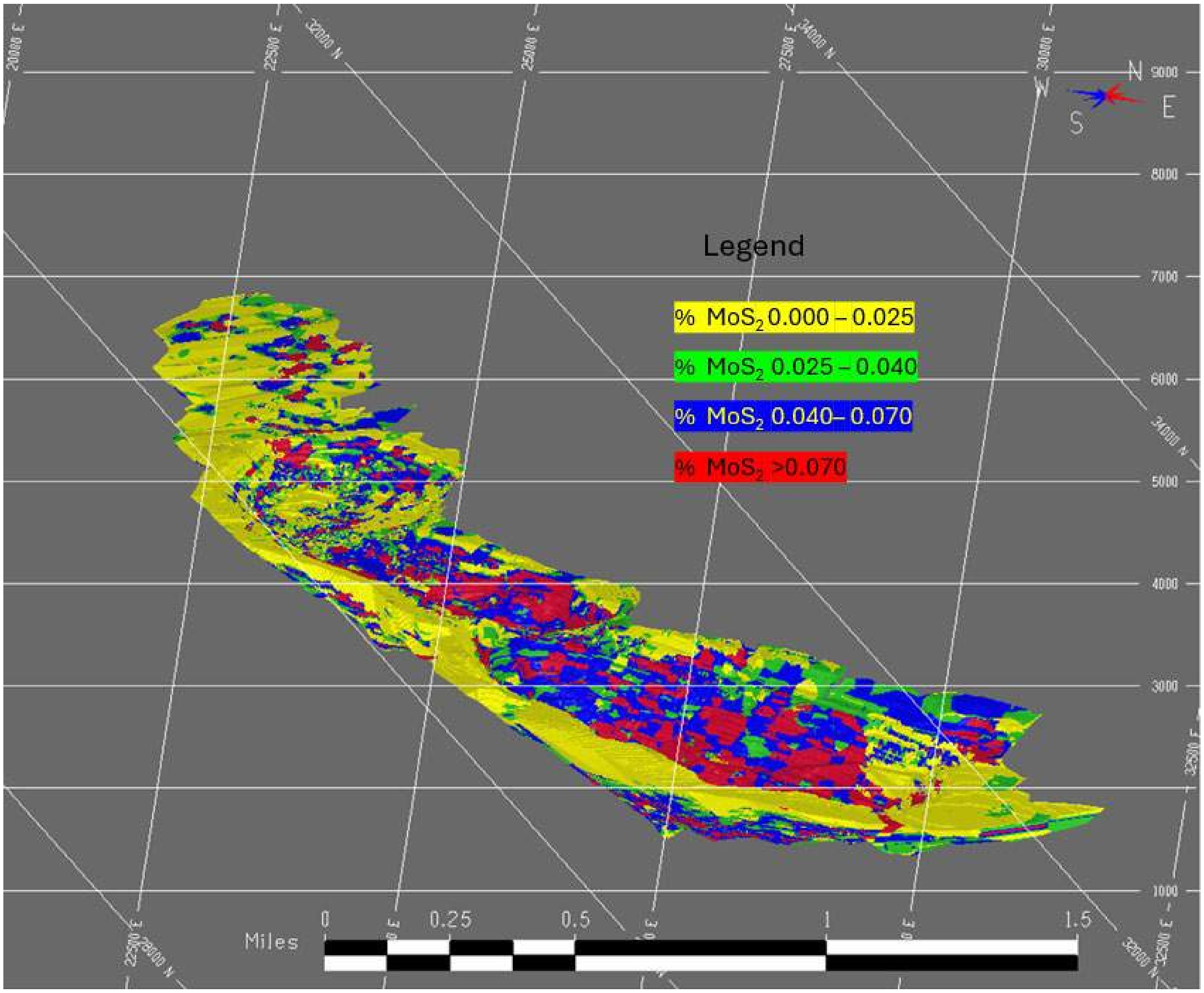


Figure 14.57 Non-Orthogonal View of Resource by Cut-off Grade
Source: *AMPL, 2025*

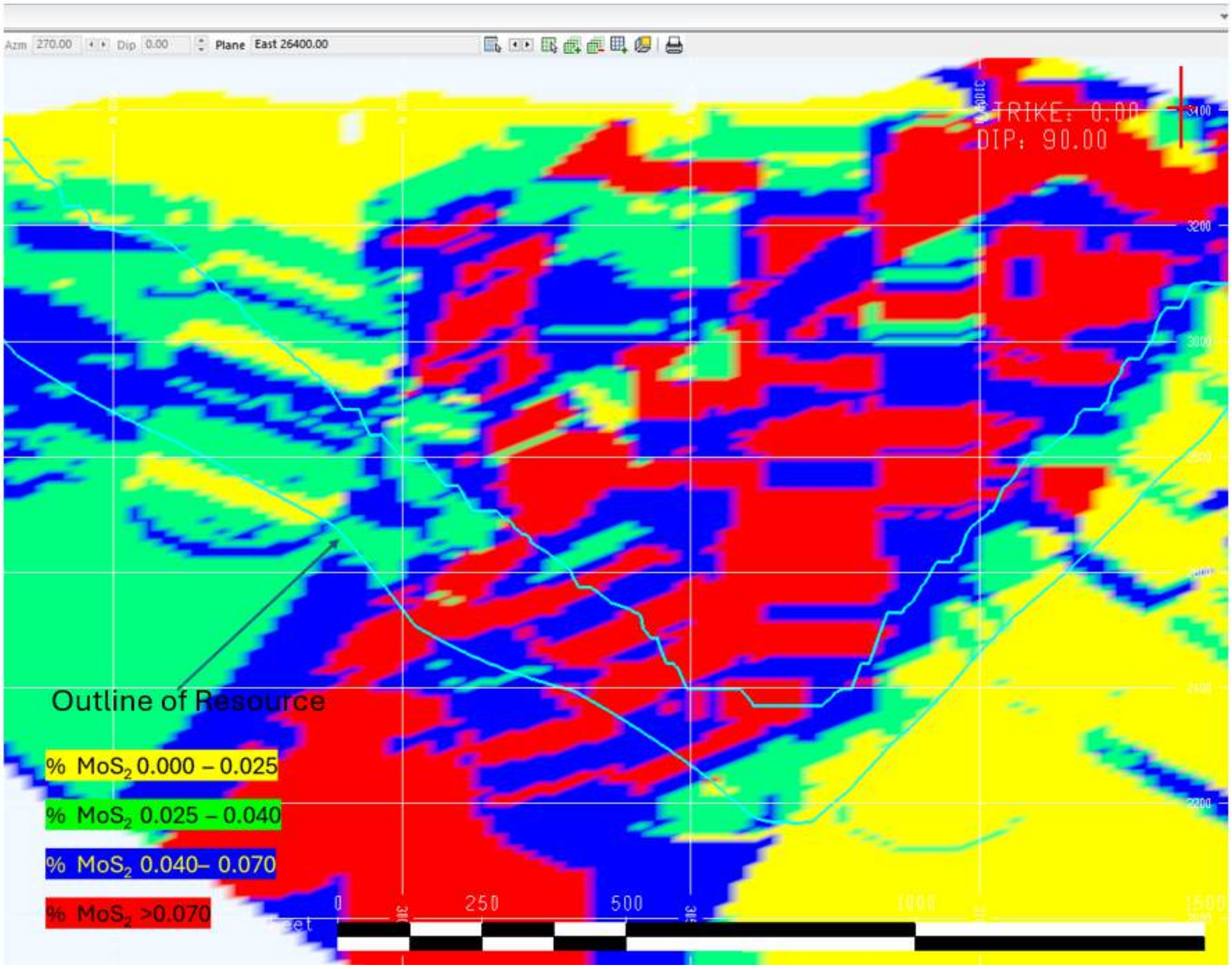


Figure 14.58 Section 26400 East View of Resource by Cut-off Grade
Source: *AMPL, 2025*

14.15 COMPARISON TO PREVIOUS RESOURCES

In September 2011, a Resource was prepared by Marek J., September 12, 2011, *NI 43-101 Technical Report, Endako Molybdenum Mine*, Independent Mining Consultants Inc. (IMC), 103 pages. A direct comparison was difficult due to differences in reporting.

This was not easily done as different pit constraining shells were employed – *i.e.*, 0.042% MoS_2 for resource in 2021 as cut-off grades changed dramatically during this period – ranging from 0.025% MoS_2 to 0.042% MoS_2 In addition, it was felt that conventional pit design would not work on an existing pit. Different areas of pits used different slope designs as well as varying bench heights. Furthermore, mining and diamond drilling occurred during the period up until December 2014. Some of the pit design could best be described as a salvage operation.

IMC did not report a Resource, which included the Reserve. It was necessary to add Proven and Probable Reserves to the Measured and Indicated Resource. This is technically not correct or precise since the dilution and recovery used to calculate the Reserve is not known. It was also necessary to remove 2.5 years of production from the total. From the production records, this was determined to be approximately

28 million tonnes at 0.076 MoS_2. It is also noteworthy that the Casey Zone was not included in the Resource statement.

A comparison was also made to an unpublished draft report *Technical Report on the Endako Mine British Columbia, Canada*, Centerra Gold Inc, NI 43-101 report (see Table 14.35, below).

TABLE 14.35
ENDAKO RESOURCES COMPARISON TO PREVIOUS RESOURCE STATEMENTS

Classification	2011 (0.030 cut-off) IMC 2011		2018 (0.042 cut-off) Centerra (WGM)		2025 (0.040 cut-off) AMPL	
	Tonnes (MIL)	MoS_2	Tonnes (MIL)	MoS_2	Tonnes (MIL)	MoS_2
Measured	130.7	0.075	67.8	0.075	100.7	0.079
Indicated	204.0	0.071	186.0	0.071	235.0	0.069
Measured + Indicated	334.7	0.072	253.0	0.072	335.7	0.072
Inferred	49.3	0.025	78.0	0.067	60.1	0.054
Notes:	Excludes Casey		Excludes Casey		Includes Casey	
	Includes 28.0 million tonnes subsequently mined					

Source: *AMPL, 2025*

14.16 RECONCILIATION TO REPORTED MINED TONNAGE

Endako Sources report a total production of 666,258,000 tonnes of ore and waste being mined. They report 391,795,000 tonnes of ore being milled with a grade of 0.136% MoS_2 (see Table 14.36, below).

TABLE 14.36
ENDAKO PRODUCTION RECORD

	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985
Tonnes Ore and Waste Mined (000s)	3,817	6,532	13,620	8,661	11,623	13,199	14,133	9,579	12,213	12,080	19,944	19,881	18,457	23,120	8,511	24,645	22,342	8,655	0	0	0
Strip Ratio	0.88	0.84	1.22	0.45	0.33	0.44	0.72	0.65	0.59	0.77	1.33	1.33	1.03	1.17	0.79	1.22	1.13	2.01	0	0	0
Tonnes Milled (000s)	2,035	3,549	6,148	5,983	8,733	9,233	8,209	4,881	7,661	6,810	8,600	8,519	9,083	10,655	4,767	11,101	10,491	2,880	0	0	0
Head Grade % MoS_2	0.183	0.237	0.212	0.178	0.189	0.182	0.162	0.149	0.146	0.165	0.161	0.163	0.161	0.135	0.129	0.141	0.110	0.091			
Recovery %	81.3	81.6	81.0	85.9	86.0	82.4	81.6	81.2	80.0	81.1	83.0	81.9	78.8	73.5	73.2	77.7	76.9	78.6			
Mo Production (000's kgs)	1,817	4,119	6,334	5,489	8,512	8,308	6,511	3,544	5,366	5,465	6,894	6,823	6,914	6,343	2,701	7,297	5,325	1,236	0	0	0

	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006
Tonnes Ore and Waste Mined (000s)	2,352	7,956	10,154	15,579	20,764	22,744	22,516	18,463	19,916	19,220	21,349	19,319	19,859	7,964	14,752	11,120	11,984	14,935	14,459	17,089	21,283
Strip Ratio	0.6	0.69	0.48	0.68	1.14	1.38	1.32	0.94	0.92	0.82	1.01	1.08	0.99	0.78	0.57	0.42	0.26	0.55	0.68	1.29	1.12
Tonnes Milled (000s)	1,466	4,716	6,845	9,265	9,703	9,543	9,701	9,585	10,385	10,430	10,023	9,377	9,805	8,426	9,396	9,386	9,641	9,706	9,350	9,604	9,700
Head Grade % MoS_2	0.147	0.159	0.134	0.127	0.134	0.137	0.150	0.145	0.133	0.140	0.141	0.142	0.128	0.108	0.111	0.128	0.122	0.111	0.112	0.099	0.115
Recovery %	79.2	83.0	83.9	82.9	83.2	82.9	80.1	79.6	74.8	74.7	77.4	76.5	76.7	76.7	76.2	78.2	75.7	80.0	79.2	77.2	78.4
Mo Production (000's kgs)	1,024	3,735	4,617	5,853	6,491	6,503	6,994	6,637	6,199	6,544	6,563	6,112	5,788	4,191	4,740	5,643	5,326	5,185	4,971	4,380	5,253

	Oct _Dec 2006	2007	2008	2009	2010	2011	2012	(July YTD)	Totals
Tonnes Ore and Waste Mined (000s)	4,416	17,277	20,376	18,251	21,149				**666,258**
Strip Ratio	0.76	0.73	0.53	0.83	0.93				**0.82**

	Oct _Dec 2006	2007	2008	2009	2010	2011	2012	(July YTD)	Totals
Tonnes Milled (000s)	2,279	9,808	10,768	9,759	10,176	10,652	14,711	8,253	**391,795**
Head Grade % MoS_2	0.099	0.100	0.116	0.098	0.100	0.090	0.070	0.080	**0.136**
Recovery %	75.7	72.8	77.7	78.4	74.5	74.0	62.7	69.4	**78.5**
Mo Production (000s kgs)	1,028	4,275	5,799	4,517	4,553	4,233	3,873	2,747	**242,775**

Source: *WGML, 2018*

AMPL utilized the “original topographic” surface as supplied by Endako, to compare reported tonnes and grade to those calculated by HxGN Mine Plan™ (see Table 14.37, below).

TABLE 14.37
ENDAKO PRODUCTION RECORD COMPARED TO CALCULATED BY HXGN MINE PLAN™

	HxGN Mine Plan™ 3D		Endako		Difference	
	Tonnes	% MoS_2	Tonnes	% MoS_2	Tonnes	
Mined	707,937,000	0.081	666,258,000	Not Reported	(41,679,000)	-6%
Milled	334,223,000	0.136	391,795,000	0.136	57,572,000	15%

Source: *AMPL, 2025*

The estimate of “mined” tonnes was based on the historical topography file, which was built on ground surveys and drill hole collars. A mill head cut-off of 0.08% MoS_2 was used as it roughly corresponds to historical cut-off grades. When all factors, including relatively imprecise survey methods, changing cut-offs, material sent to stockpiles and the actual ore waste contacts while mining are considered, the correlation is consistent with expectations and validates the modelling.

15.0 MINERAL RESERVES

The Endako Mine presently has no Mineral Reserves. Mineral Reserves can only be determined by a Pre-Feasibility or Feasibility Study.

16.0 MINING METHOD

The past producing Endako Mine utilised conventional open pit hard rock mining techniques. Ore and waste were drilled, blasted, loaded with electric wire rope shovels or a front-end loader and truck and hauled to either an in-pit crusher, waste stockpile or ore stockpiles. The present proposed Endako Mine would be mined by expanding existing and mining new open pits, similar to past operations.

16.1 GEOTECHNICAL

A detailed geotechnical and hydrological assessment was performed by WSP-Golder in 2022. The main results of this WSP-Golder study are presented in this section.

The assessment included the re-activation of the Denak East and Endako open pits, as well as mining a proposed Northwest Extension open pit, to the north of the Denak West pit, as shown in Figure 16.1, below.

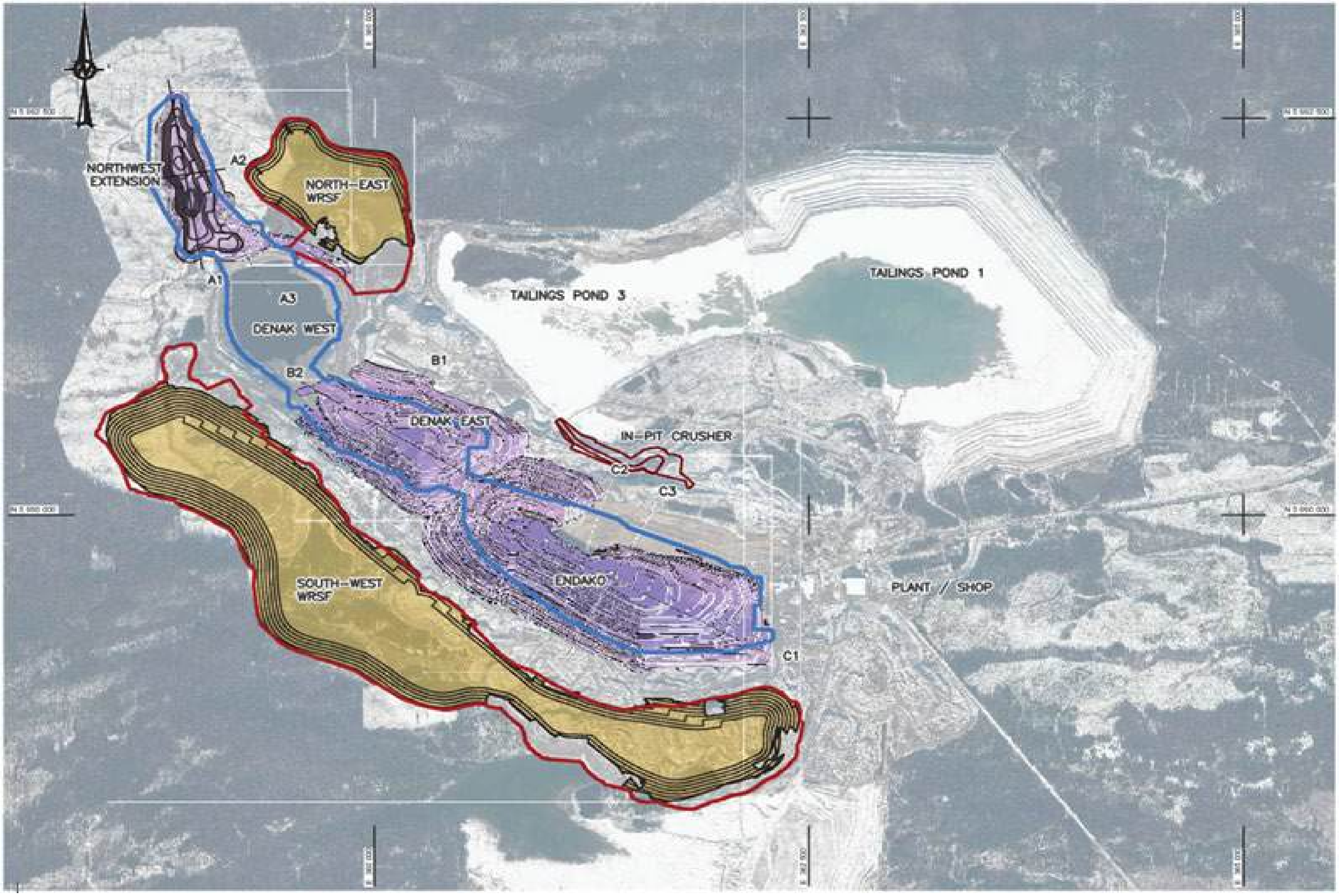


Figure 16.1 LOM Open Pit Plan for Endako Mine
Source: *AMPL 2025*

The stability of the previously proposed ultimate pit configurations was assessed by WSP-Golder (Golder 2010, 2014, 2014a, 2014b, 2015).

The discussion of open pit slope stability and the recommended open pit slope designs are based on the results of the previous geotechnical investigations, stability assessments, and experience with the open pit slope stability performance in the existing open pits.

16.2 ENDAKO PIT

The recommended bench configurations for the Endako open pit are summarised in Table 16.1, below. The pit will be excavated to an ultimate elevation of approximately 628 m (2,060 ft), compared with the current pit crest at 982 m (3,220 ft), resulting in a total depth of about 386 m. The current pit depth is 750 m (2,460 ft) , meaning a further 130 m will be excavated over an 11-year period.

TABLE 16.1
RECOMMENDED DESIGN BENCH CONFIGURATIONS – ENDAKO OPEN PIT

Proposed Wall	Proposed Wall Facing Azimuth	Catch-bench Width (m)	Bench Height (m)	Bench Face Angle (degrees)	Inter-ramp Angle (degrees)
South Wall[1]	320° to 055°	10.5	13.4	70	**41**
West Wall (above West Basalt Fault)	060° to 120°	12.9	13.4	65	**35**
West Wall (below West Basalt Fault)	060° to 120°	12.2	26.9	70	**50.7**
North Wall (West of East Boundary Fault to West Edge of Instability Zone)[2]					
Above 2,838 ft Bench	125° to 220°	9.6	13.4	65	**40**
Above Ramp		8.5	13.4	70	**45**
Below Ramp		11	26.9	70	**52.3**
North Wall					
Above Ramp	125° to 220°	8.6	13.4	70	**45**
(West of Instability Zone)					
Below Ramp		11	26.9	70	**52.3**
North Wall (East of East Boundary Fault)	180° to 220°	11	26.9	70	**52.3**
East Wall	**225° to 315°**	**9.1**	**13.4**	**65**	**41.1**

Notes:
[1]Recommended bench design configuration for the south wall as provided in Golder 2010. These recommendations are considered appropriate for the currently proposed Endako open pit design. A 24 m wide geotechnical berm was recommended and included at elevation 892 m.
[2]The west edge of the instability zone is located at approximately 27,000 East.

The assessment considers a new layout configuration with two access ramps exiting to the west and east sides of the Endako open pit (Figure 16.1). A complementary stability assessment was prepared to evaluate the stability of the northwest portion of the north wall (Golder, 2021).

The Endako and the Denak East open pits are separated by the West Basalt Fault and the Denak Fault. These faults are wide zones of poor-quality rock that dip toward the northwest at shallow angles. A waste saddle would separate the two pits and the transition of the north and south walls of the Endako, and Denak open pits would be excavated through these faults. The ultimate Endako open pit would be accessed via two haul roads from the west and east sides of this pit. The western haul road would progress upward to the east side of the adjacent Denak East open pit and connect to the haul road that would access the crusher. An east-west trending slot cut would be excavated through the waste saddle, exposing poor-quality rock associated with faults on the north and south sides of the slot cut. A stability assessment was completed to evaluate the impact of the fault zone with respect to the slot cut (Golder, 2014a). The assessment recommended that within the slot cut, the walls along the southeast and east sides of the open pit should be laid back to an angle of approximately 35 degrees to minimise undercutting and planar failure within the fault complex.

16.3 DENAK EAST PIT

The recommended bench configurations for the Denak East open pit are summarised in Table 16.2, below (Golder, 2010). The pit will be excavated to an ultimate elevation of approximately 762 m (2,500 ft). Its current depth is 854 m (2,800 ft), meaning an additional 100 metres will be excavated over an 11-year period.

TABLE 16.2
RECOMMENDED DESIGN BENCH CONFIGURATIONS – DENAK EAST OPEN PIT

Wall Facing Azimuth	Pit Wall Sector	Bench Height (m)	Recommended BFA (degrees)	Recommended Maximum IRA (degrees)	Recommended Catch-bench Width (m)
000°	165° to 195°	13.4	70	41	**10.6**
030°	195° to 225°	13.4	70	41	**10.6**
060°	225° to 255°	13.4	70	46.2	**8.0**
090°	255° to 285°	13.4	70	46.2	**8.0**
120°	285° to 315°	13.4	70	42	**10.0**
150°	315° to 345°	13.4	70	36	**13.6**
180°	345° to 015°	13.4	70	36	**13.6**
210°	015° to 045°	13.4	70	46.2	**8.0**
240°	045° to 075°	13.4	70	46.2	**8.0**
270°	075° to 105°	13.4	70	40	**10.6**
300°	105° to 135°	13.4	70	35	**12.9**
330°	**135° to 165°**	**13.4**	**70**	**41**	**10.6**

Note:
See Denak Fault/West Basalt Fault Domain design in the Endako open pit summary section.
Mining in the Denak West open pit has been completed and will not be recommissioned.

16.4 NORTHWEST EXTENSION PIT

The recommended bench configurations for the Northwest Extension open pit are summarised in Table 16.3, below (Golder, 2014b). The pit will be excavated to an ultimate elevation of approximately

829 m (2,720 ft). With the current depth at 860 m (2,820 ft), this represents only a minor additional excavation of about 30 metres.

TABLE 16.3
RECOMMENDED DESIGN BENCH CONFIGURATIONS – NORTHWEST EXTENSION OPEN PIT

Sector Wall Facing Azimuth (degrees)	Proposed Wall Sectors	Optimum Inter-ramp Angle	Catch-bench Width (m)	Bench Height (m)	Bench Face Angle (degrees)	Inter-ramp Angle (degrees)
180° to 246°	North Northeast East Southeast	47	10	13.4	70	42
246° to 267°	N/A	48	10	13.4	70	42
267° to 307°	N/A	62	8	13.4	70	46
307° to 355°	N/A	N/C[1]	8	13.4	70	46
355° to 000°	N/A	64	8	13.4	70	46
000° to 014°	N/A	56	9	13.4	70	44
014° to 019°	N/A	64	8	13.4	70	46
019° to 092°	South Access Ramp Southwest Alcove Southwest West	57	8	13.4	70	46
092° to 115°	N/A	53	9	13.4	70	44
115° to 156°	Northwest	50	9	13.4	70	44
156° to 180°	N/A	47	9	13.4	70	44

Note:
[1]N/C: No control on inter-ramp angle.

The Northwest Extension and Denak West open pits would be separated by a ridge (rock pillar) that will be about 27.5 m wide at the crest and will expand to a width of 102 m near the Northwest Extension open pit floor (Figure 16.2, below). Based on the current information, there is no interpreted fault transecting this area and connecting the two open pits.

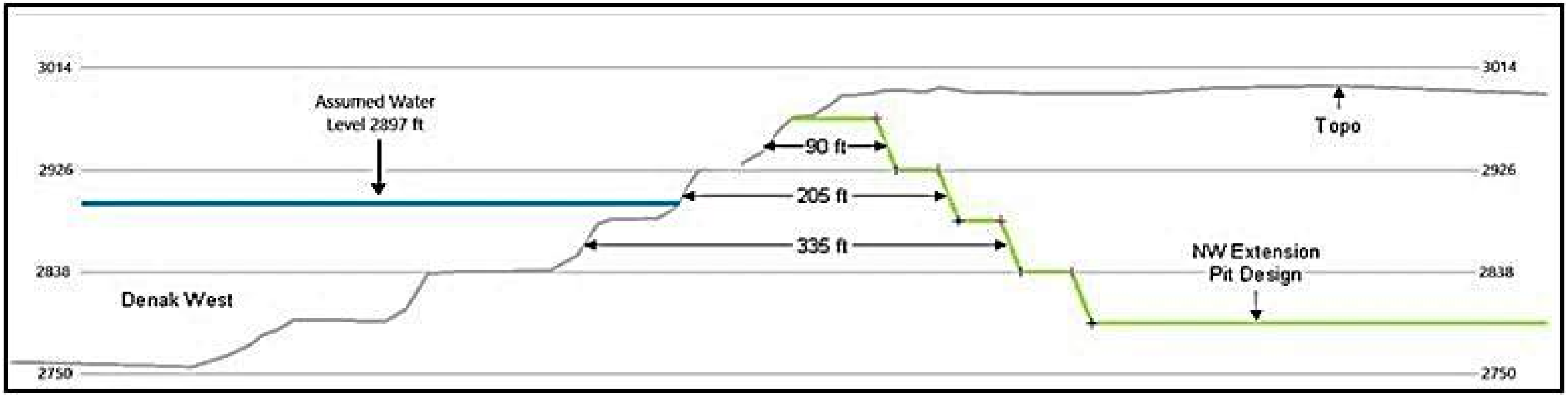


Figure 16.2 Rock Pillar Separating Denak West and Northwest Extension Open Pits
Source: *WSP-Golder, 2022*

16.5 PIT SLOPE MONITORING

Effective wall control blasting techniques (with trim shots and with or without pre-shear) will be applied for the excavation of the final walls, and for benches in sensitive areas (*e.g.*, along the ramps or where there were previous slope instabilities).

The monitoring program will include slope conformance and slope performance monitoring. The slope conformance will verify if there is good adherence between the excavated benches and the design bench configurations. It will also serve to confirm assumptions made regarding the geology. The slope performance monitoring will monitor the slope stability and detect unexpected conditions with sufficient lead time to enable the adoption of remedial measures.

The Endako Mine will develop and operate with a Ground Control Management plan, which will include training of geotechnical personnel, and a Trigger Action Response Plan (TARP) for monitoring and response actions should an instability occur. Yearly geotechnical reviews of the slope stability and the data will be conducted by a third-party geotechnical consultant selected by the Mine management.

16.6 POTENTIALLY ECONOMIC MINERALISATION – OPEN PIT OPTIMISATION

The potentially economic mineralisation was defined as those blocks falling within an optimised open pit shell derived from the economic parameters shown in Table 16.4, below. The unit costs used in the pit optimisation process were based (and converted to 2025 equivalent costs) on preliminary estimates developed by WSP-Golder based on first principles and other owner operated open pit mining costs and general knowledge of mining, processing and general and administration (G&A) costs for similar type operations. The open pit optimisation was conducted using DataMine™.

The potentially economic mineralisation was determined using a breakeven cut-off where revenue is equivalent to marginal costs. The $9.7/tonne breakeven cut-off, derived from the sum of the estimated processing, surface department and G&A costs, does not include mining costs as all material contained within a shell is considered mined and sent either to the waste dump or the processing plant.

The open pit potentially economic mineralisation (includes dilution) estimate is shown in Table 16.5, below, and may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues. The potentially economic Mineral Resources estimate, which takes geologic, mining, processing and economic constraints into account, are confined within a pit shell and are classified in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves.

TABLE 16.4
OPEN PIT OPTIMISATION PARAMETERS

Area	Parameter	Unit (metric)	Value (metric)	Unit (Imperial)	Value (Imperial)
Financial	**Price**				
	Molybdenum Price	US$/kg MoS_2	49.60	US$/lb MoS_2	22.50
	Molybdenum Price	CA$/kg	66.14	CA$/lb MoS_2	30.00
	Exchange Rate	US$:CA$	0.75	US$:CA$	0.75
	Discount Rate	% p.a.	8.00	% p.a.	8.00
	Selling Costs				
	Refining Cost	CA$/kg Mo	2.53	CA$/lb Mo	1.14
	Transportation Cost	CA$/kg Mo	0.58	CA$/lb Mo	0.26
	Processing Cost				
	Processing Cost	CA$/tonne Milled	4.90	CA$/ton Milled	4.45
	Surface Services Cost	CA$/tonne Milled	0.50	CA$/ton Milled	0.45
	Water Treatment Plant	CA$/tonne Milled	0.43	CA$/ton Milled	0.39
	Environmental Cost	CA$/tonne Milled	0.50	CA$/ton Milled	0.45
	G&A Cost	CA/$tonne Milled	1.00	CA$/ton Milled	0.91
	Mining Cost				
	Ore Mining Cost	CA$/tonne	2.60	CA$/ton	2.36
	Waste Mining Cost	CA$/tonne	2.60	CA$/ton	2.36
Process Parameters	Processing Rate	Mtpa	26.90	M ton pa	29.64
	Processing Recovery	%	20.675 × MOS_2 %^2 + 5.0835 × MOS_2 % + 0.4852	%	20.675 × MOS_2 %^2 + 5.0835 × MOS_2 % + 0.4852
	Refining Recovery	%	99.70	%	99.70
	Transport Losses	%	0.50	%	0.50
	Contained Mo in MoS_2	%	59.94	%	59.94
	Concentrate Grade	MoS_2 %	91.50	MoS_2 %	91.50
Mining Parameters	Mining Rate	Mtpa	26.90	M ton pa	29.64
	Mining Recovery	%	3.00	%	3.00
	Mining Dilution	%	Block Model ORE% Field	%	Block Model ORE% Field
	Cut-off Grade	MoS_2 %	0.04	MoS_2 %	0.04

TABLE 16.5 OPEN PIT MINEABLE RESOURCES INVENTORY		
Mineral Resource Category	**Tonnes (Mts)**	**Grade (MoS_2 %)**
Measured	187.5	0.077
Indicated	79.6	0.072
Inferred	8.57	0.057
Total	**275.66**	**0.074**

Note:

1. Mineral Resources are not classified as Mineral Reserves. Mineral Reserves are only designated once a feasibility study is conducted, demonstrating economic viability.
2. Modifying factors, such as mining recovery and dilution, were applied accordingly. A mining recovery factor of 3% ore loss was used. For mining dilution, this was simulated by re-blocking the resource model from a block size of 25 × 25 × 10 m to 50 × 50 × 44 m, to reflect potential dilution resulting from drill-and-blast and load-and-haul activities associated with this style and scale of open pit mining.
3. A cut-off grade of 0.04% MoS_2 was applied.
4. The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature, and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured mineral resources. It is also uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category. The lower in-situ grade means that, after mining dilution and recovery are applied, much of this material may report as mineralised waste and will not be considered mineable. However, it may be considered treatable in the future, towards the end of the mine's life, should the pricing and economics change.

Plan views by year of the open pit shell production are presented in Figure 16.3, below. The open pit undiluted Mineral Resources estimate was prepared using the DataMine™ open pit optimisation software, and the 2025 AMPL geological block model for the Endako deposit.

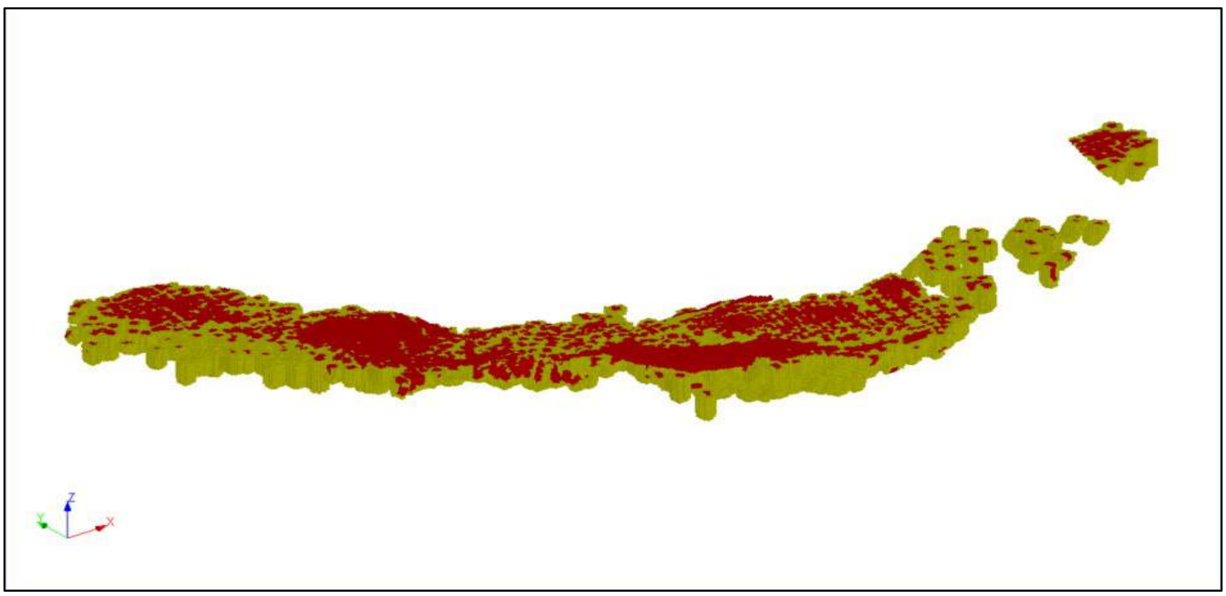


Figure 16.3 Mineral Resource Classification Geospatial Representation
***Source:** AMPL, 2025*

As shown in the Figure 16.4. below, most of the highest-classified material within the pit is located in the upper levels, with a significant proportion of Measured Resources distributed across the Endako Project, including Endako, Denak East and Denak West. This distribution is expected and reflects the higher data density in these areas. As operations progress, further drilling is anticipated to convert Inferred and Indicated material into Measured Resources as pit depth increases.

Figure 16.4 Pit Shell Aerial View (LOM pit crest outline shown in white), Based on a Google Earth™ Image Current as of 30 April 2025

Source: *Google Earth™, 2025*

16.6.1 All Pits Combined

The combined optimisation results for all three pits are shown in Figure 16.5, below. The Revenue Factor (RF) of 0.88, based on the preferred pit No. 79, is considered the most optimal.

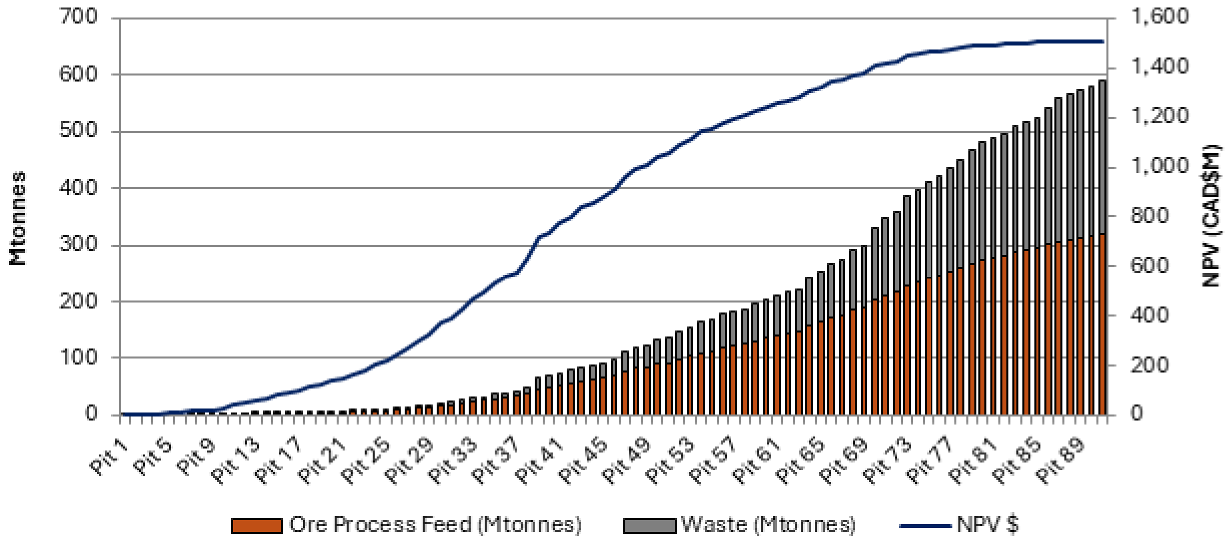


Figure 16.5 Optimisation Tonnage and NPV Results
Source: *AMPL, 2025*

This conclusion reflects several factors, including - but not limited to - NPV, volume and strip ratio, as well as compliance with certain physical and practical constraints outlined in Figure 16.6, below. Beyond an RF of 88%, the law of diminishing returns becomes increasingly evident under current pricing assumptions.

The RF range assessed was 8% to 100%, based on a base-case price assumption of CA $66.14/kg of molybdenum.

The open pit optimisation results are presented in Table 16.6, below.

TABLE 16.6
OPTIMISATION RESULTS

Item	Unit	Amount
Pit	n°	79
Price Factor	%	88
Total Rock	Mt	467
Ore Feed[1]	Mt	271
Waste	Mt	200
Strip Ratio	t:t	0.75
MoS_2 Diluted	%	0.075
Tcc_91.5%	t	158,901
Mo Oxide Concentrate Contained Metal	t	87,149
NSR (CA$ million)	$ million	5,078
NPV (CA$ million)	$ million	1,488

Note:
[1]The mining recovery and dilution factors applied were 95% and 5%, respectively.

The ultimate open pit design cross sections are presented in Figure 16.7 to Figure 16.8, below presenting Endako, Denak East and Denak West open pits.

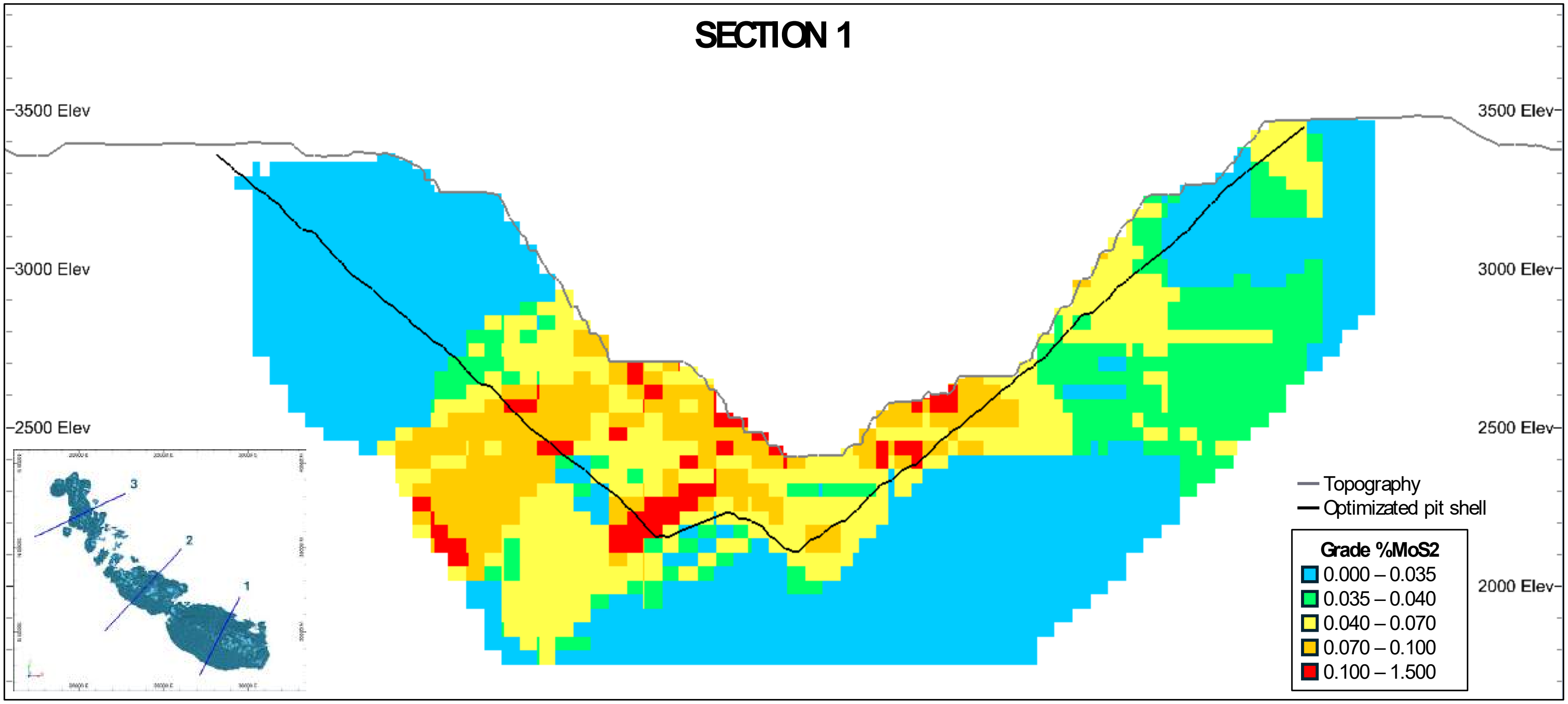


Figure 16.6 Ultimate Open Pit Design Cross-section 1 – Endako
Source: *AMPL, 2025*

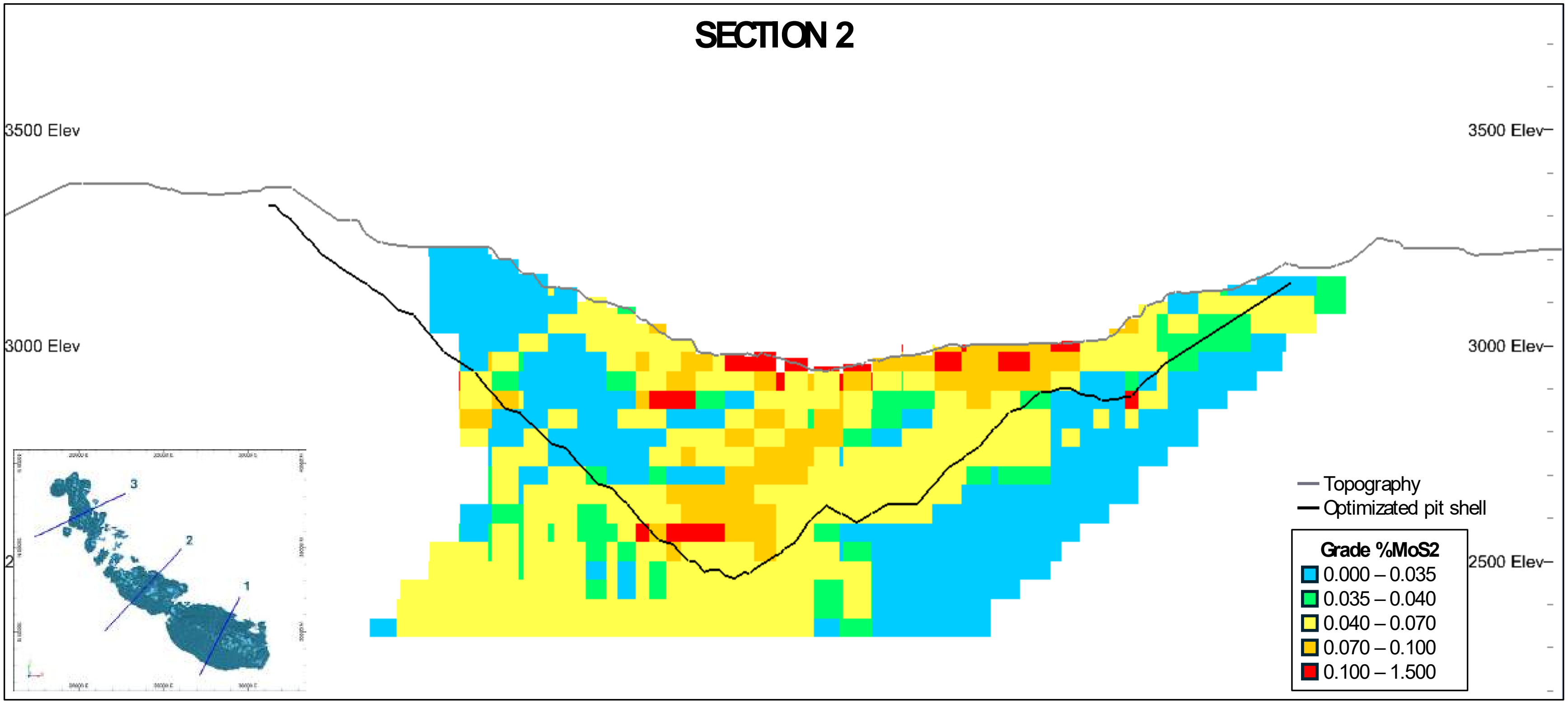


Figure 16.7 Ultimate Open Pit Design Cross-section 2 – Denak East
Source: *AMPL, 2025*

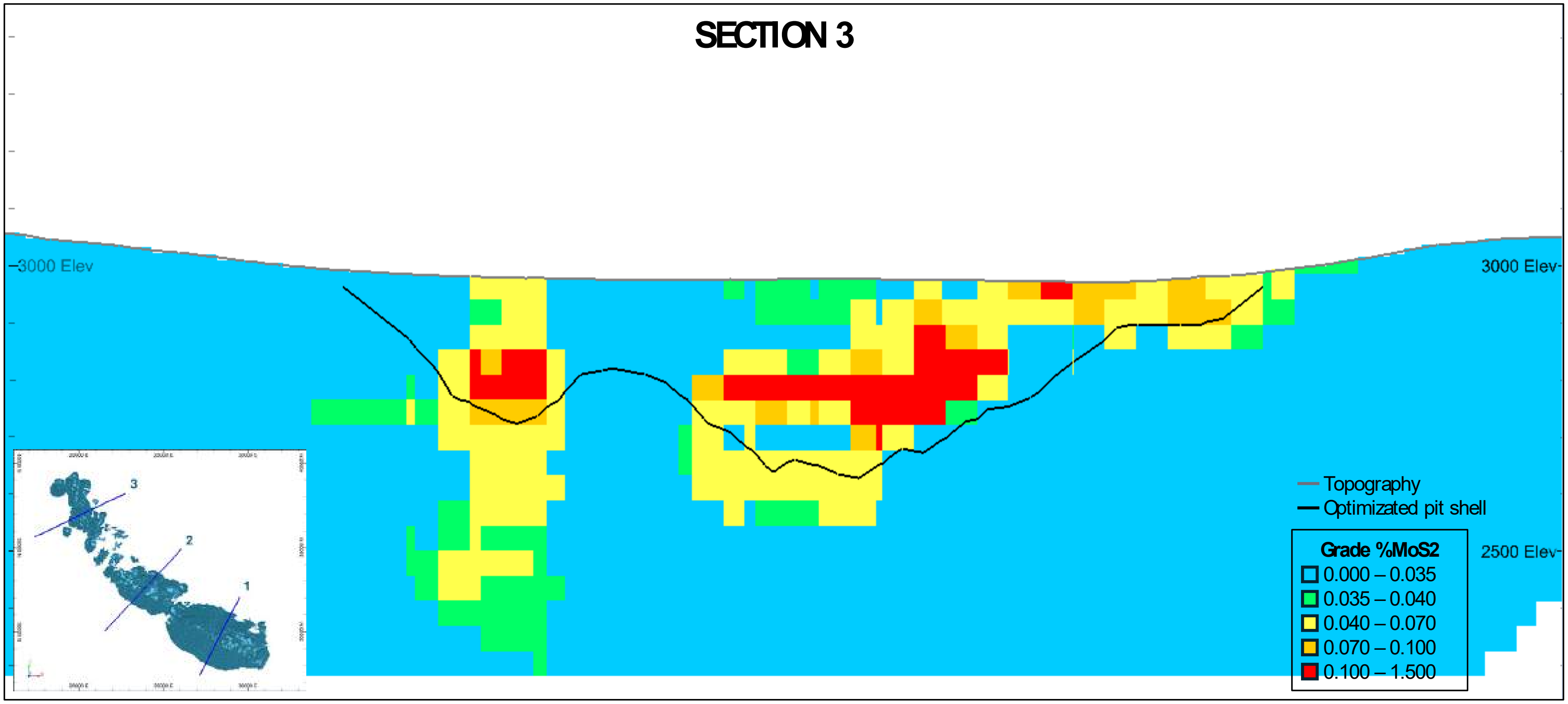


Figure 16.8 Ultimate Open Pit Design Cross-section 3 – Denak West
***Source:** AMPL, 2025*

16.6.2 Existing Haul Roads

Current haul road designs applied by AMPL are 36.6 m wide and based on the WSP-Golder recommendations. The haul roads have been designed following the guidance of the Mines Act BC, *Health, Safety and Reclamation Code for Mines in British Columbia*.

Most haul roads are designed as double-lane, although single-lane roads may also be implemented. A design based on standard assumptions is shown in Figure 16.9, below. The double-lane sections of the haul ramp were designed to accommodate 3.4 times the canopy width of the widest haulage vehicle used on the road – a 218 to 240-tonne class haul truck, comprising a mix of Terex/Bucyrus MT 4400 and CAT 793F models – with additional clearance for a berm and ditch.

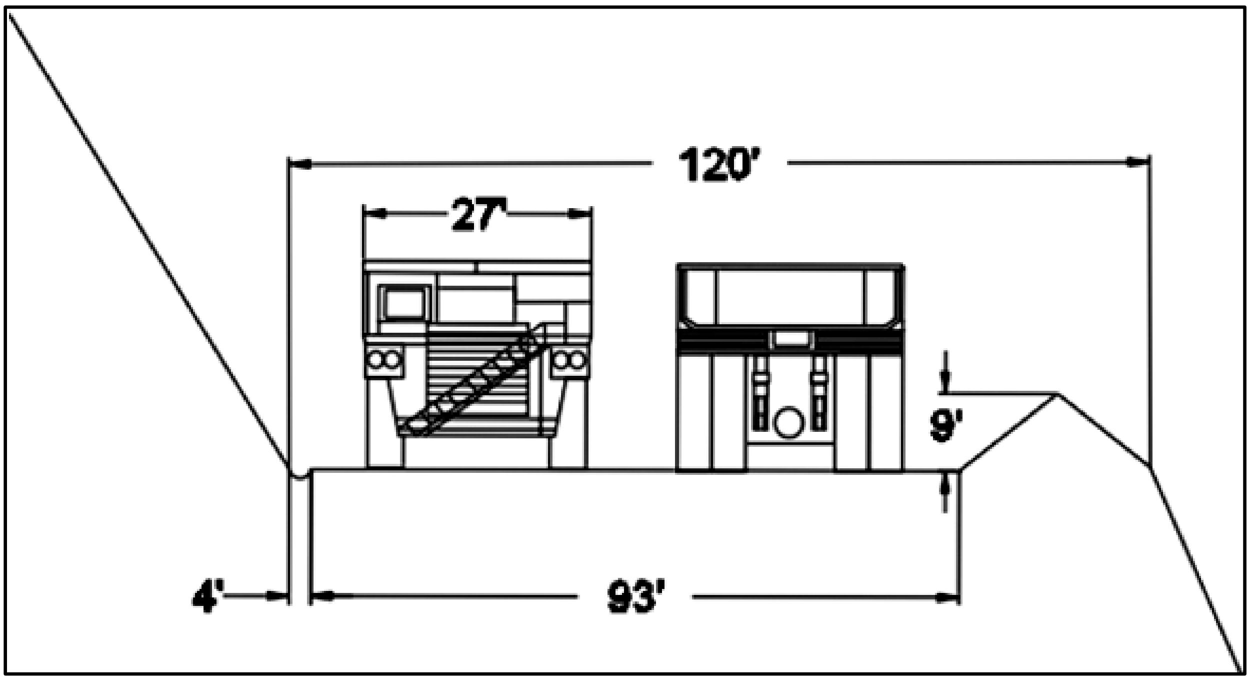


Figure 16.9 Haul Road Design Dimensions
Source: *AMPL, 2025*

Single-lane sections were designed to accommodate twice the width of the haul trucks. A berm of approximately 2.7 m was positioned and maintained along the pit-side edge of the haul road wherever a drop-off greater than 3 m occurs.

The berm is designed at a slope ratio of 1.3:1, which is equivalent to ¾ of the tire height of a 240-tonne haul truck (CAT 793). Breaks that do not exceed the width of the blade of the equipment constructing and maintaining the breaks are incorporated. The width of the barrier is excluded from the travel width. A 1.2 m wide ditch was also included on the inside of the haul ramp to allow for drainage of surface water and snow clearance. The total width of the two-lane ramp was calculated to be 36.5 m, and the total width of the single-lane ramp was calculated to be 24.4 m. For vehicle runaway protection, on roadways where the grade exceeds 5%, there would be runaway lanes, or retardation barriers, where conditions/risk warrant.

16.6.3 Mining Operations

Pre-production work would include establishing the main haul road to the primary crusher building and surface road to the mine facilities, including explosives magazines.

The open pit will be mined using conventional open-pit equipment and technologies. Mineralised material and waste rock will be blasted, excavated, loaded and hauled either to the primary crusher or to the waste rock management area. It is assumed that the owner will develop and operate the open-pit mines.

Equipment will be sized according to the pit configurations and a planned mining rate of 45 million tonnes per annum (ore and waste combined). Bench heights of 6 m are envisaged, utilising standard mining equipment, such as track-mounted drills, wire rope shovels, wheel loaders, 218-240-tonne haul trucks and bull dozers. The open pit is expected to operate 350 days per year, and the mining fleet will be sized accordingly.

This study utilises owner operated crews with leased major mining equipment and purchased smaller support equipment.

16.6.4 Waste Rock Storage Facilities (WRSF)

The existing waste rock storage facilities (WRSF) will be expanded and new ones added, in future operations, within the current mine boundaries.

Mined waste rock is to be consigned to the proposed northeast WRSF and the southwest WRSF. The proposed northeast WRSF has been designed to accommodate waste rock that is mined from the Denak East open pit and from the Northwest Extension open pit. The proposed southwest WRSF will be located along the south side of the open pits. A general site layout with the disposition of both WRSFs can be seen in Figure 16.2 below. At the end of the LOM, the northeast WRSF will be at capacity, and the southwest WRSF will have 105 million tonnes of additional capacity.

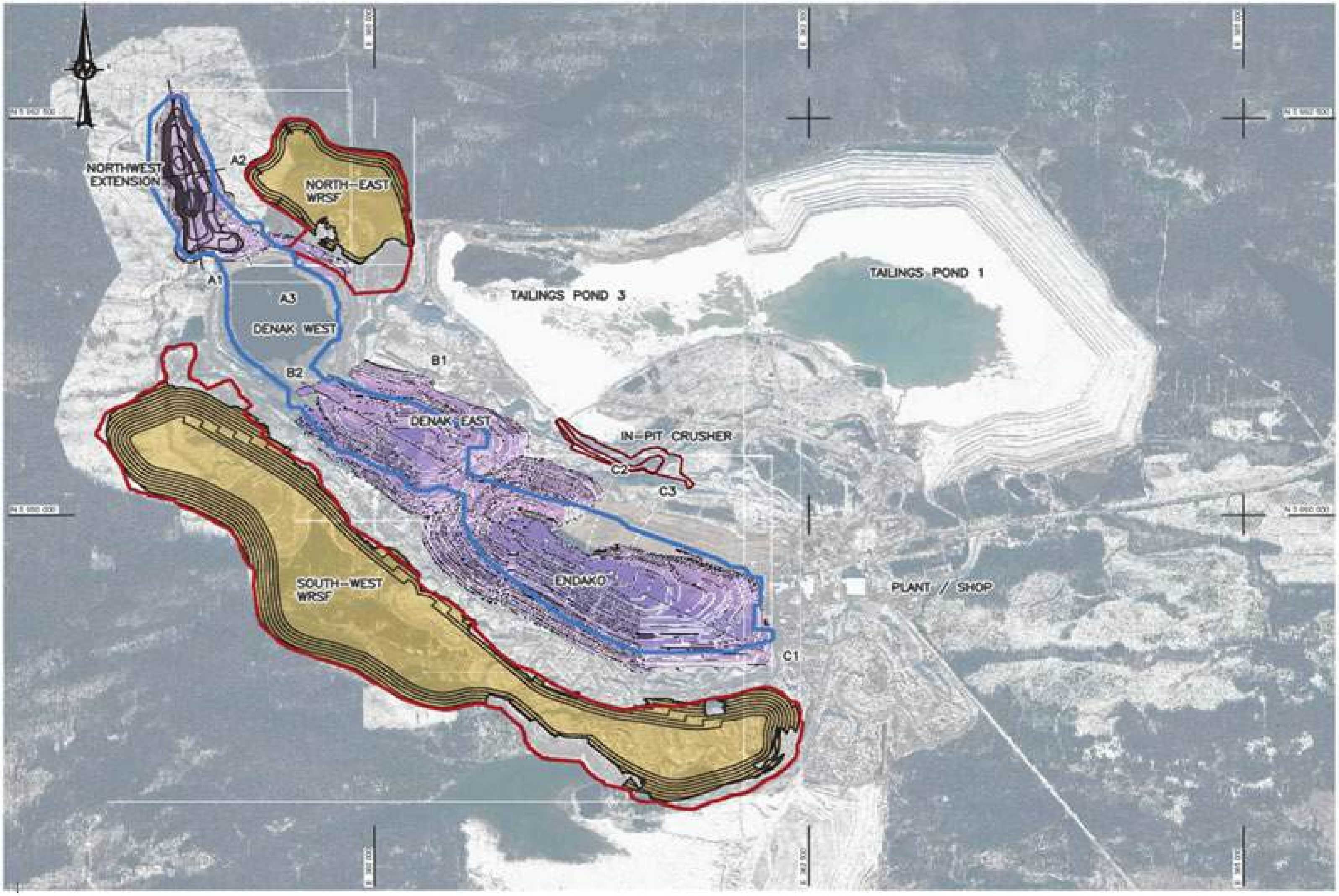


Figure 16.10 LOM Open Pit Plan for Endako Mine
Source: *AMPL 2025*

16.6.5 Acid Generating Rock

The definition of acid generating rock (Potentially Acid Generating "PAG" and Non-Acid Generating "NAG"), assessed by SRK in 2014, defines the following three categories of rock:

- Non-PAG (NAG) = Neutralisation potential (NP) / acid production (AP) greater than (>) 2
- Low Reactive PAG (PAG) = NP/AP between 1 and 2
- Highly Reactive PAG (HR PAG) = NP/AP less than (<) 1

Acid generating rock for the site is concentrated in the pyrite zone in the south wall of the Endako and Denak East open pits. The Amended Permit, described in Item 20 in the TR, calls for all rock that could be PAG to be backfilled into a completed open pit and submerged under water. NAG waste material will be placed either in the north or southwest WRSF, depending on the nearest haul distance and location capacity.

AMPL acknowledges and agrees with Golder's previous recommendations to conduct a detailed review of the PAG and NAG classifications, as this directly impacts the amount of material assumed to be placed back in the open pits, which in turn affects water storage capacity.

16.6.6 Pit Dewatering and Water Management

Recent satellite imagery clearly shows that the main pits are flooded. A campaign to actively dewater the pits is, therefore, required. This will most likely involve pumping and recirculating water between pits, a process that must be carefully coordinated with the mining schedule and the anticipated Project restart dates.

This activity will be addressed in greater detail in the Hydrogeology and Environmental Management chapters as part of the more detailed feasibility studies and reporting. Future studies will include detailed water balance estimates incorporating water demand data from the process facility.

16.6.7 Ultimate Pit Design

For efficient mining, a minimum mining width allowing a haul truck to perform a full turning circle is recommended. Considering turning clearance radius, protective berm and an extra width allowance, a minimum mining width of 54.9 m is recommended, which allows space for electric shovels, P&H 2800, and 240-tonne haul trucks.

The ultimate pit was designed using the Whittle RF 0.88 shell generated from a molybdenum selling price of US$49.60/kg (US$22.50/lb) of molybdenum oxide.

16.6.8 Northwest Extension Open Pit

The Northwest Extension open pit was designed to leave a ridge of in-situ material between the south wall of the Northwest Extension open pit and the Denak West open pit, avoiding the need to dewater the Denak West open pit. Geological conditions in the area, such as faults, are unknown to WSP-Golder. Further hydrogeological assessment should be completed to assess potential water infiltration rates. From the Whittle analysis, three satellite open pits were produced. As these areas produce limited potentially mineable resource tonnages and would require additional permitting, the Northwest Extension was included in the mine plan but the satellite areas were omitted. The satellite open pits contain approximately

1.5 million tonnes of potentially economic mineralisation but minimum mining width considerations eliminate these areas.

16.6.9 Mining Schedule

Mining activities have been planned and scheduled to include pre-stripping of waste rock and the extraction of potentially mineable mineralisation, alongside waste rock removal throughout the LOM.

AMPL has adopted a ROM mining rate of 26.9 million tonnes per annum of potentially economic mineralisation.

The maximum total mining rate is 48 million tonnes per annum, including waste, with an average of 45 million tonnes per annum over the LOM. This equates to approximately 1.85 million tonnes per annum of contained MoS_2 on average, with production peaking at between 2.35 and 2.40 million tonnes per annum in Year 3. High-grade material within the open pits is prioritised for early extraction, while material grading between 0.035% and 0.04% MoS_2 will be stockpiled for later treatment during the mine life.

The LOM strip ratio is favourable at 0.70 tonnes of waste per tonne of potentially mineable mineralisation. The peak strip ratio occurs in Year 1 at 1.1, then stabilises at approximately 0.65.

The Endako Mine schedule is based on the optimised open pit shell, with mining recovery and mining dilution rates of 95% and 5%, respectively.

The total tonnes of material mined from the designed open pit are expected to add marginally to the strip ratio (see Table 16.7, below).

TABLE 16.7
LIFE-OF-MINE PRODUCTION SCHEDULE
(TONNES/MILLION)

Period	Unit	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Endako	Ore (Mt)	**153.86**	1.06	0.11	1.74	23.21	22.91	18.25	5.39	26.97	26.54	22.54	5.14
	MoS_2 (%)	**0.072**	0.060	0.096	0.056	0.068	0.074	0.073	0.068	0.072	0.074	0.073	0.077
Denak	Ore (Mt)	**113.85**	19.76	26.74	24.48	3.93	4.09	8.81	21.61		0.36	4.08	
	MoS_2 (%)	**0.080**	0.079	0.078	0.092	0.105	0.067	0.072	0.074		0.061	0.072	
Casey	Ore (Mt)												
	MoS_2 (%)												
Total Ore	**(Mt)**	**267.71**	**20.82**	**26.85**	**26.22**	**27.14**	**27.00**	**27.06**	**27.00**	**26.97**	**26.90**	**26.62**	**5.14**
MoS_2 Grade	(%)	**0.075**	0.078	0.078	0.089	0.074	0.073	0.073	0.073	0.072	0.074	0.072	0.077
Endako Waste	(Mt)	**103.32**	2.33	0.01	7.52	14.22	9.68	15.04	13.54	18.08	13.36	9.34	0.19
Denak Waste	(Mt)	**71.25**	21.26	17.61	11.67	2.09	6.84	3.87	5.46		1.61	0.85	
Other Waste	(Mt)	**2.89**	0.02	-	-	1.08	1.76	0.02	-	-	-	-	0.00
Deposits	(Mt)	**8.68**	1.57	1.53	0.59	1.47	0.73			0.96	1.82		
Total Waste	**(Mt)**	**186.14**	**25.19**	**19.16**	**19.78**	**18.86**	**19.01**	**18.93**	**19.00**	**19.04**	**16.80**	**10.18**	**0.19**

Note:

1. The mining schedule is based on the optimised pit shells. The expected and inventory and strip ratio may vary once detailed mine-design has been completed.
2. The mine schedule shown in Table Mining Method.5 excludes Inferred Mineral Resources which equates to approximately 8.6 million tonnes.
3. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that the results indicated in the PEA will be realised.
4. Excludes Casey mineralisation.
5. Cut-offs used are 0.04 (primary cut-off).
6. Marginal ore between 0.035 to 0.04 is stockpiled and treated later in the mine-life.

Figure 16.11 and Figure 16.12, below, provides a visual breakdown of the ore production by open pit and year.

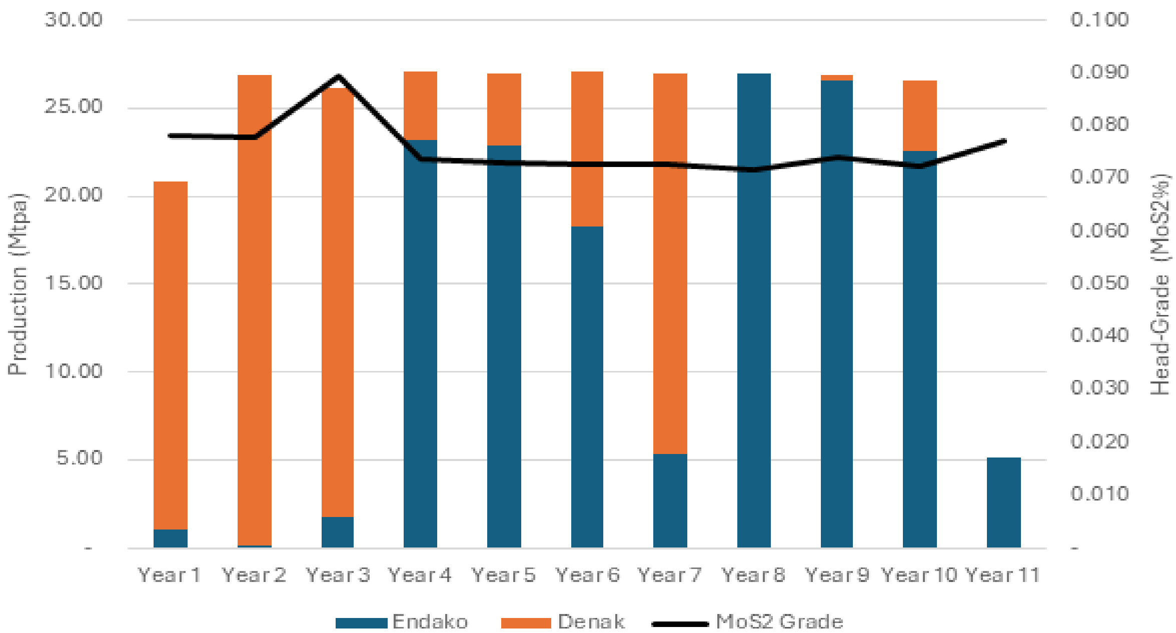


Figure 16.11 Ore Production Breakdown by Pit/Area (excludes Inferred)
***Source:** AMPL, 2025*

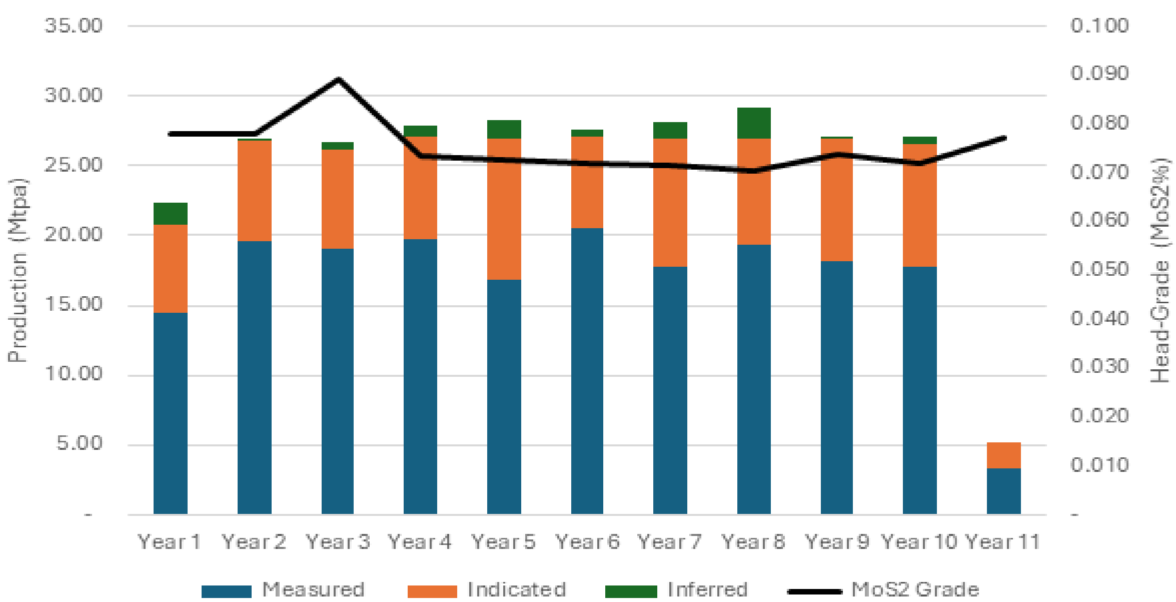


Figure 16.12 Production Schedule - Breakdown by Material Classification
***Source:** AMPL, 2025*

Assuming an average mining rate of 46 to 48 million tonnes per annum (potentially economic mineralisation and waste) over the LOM, and a processing rate of approximately 26.9 million tonnes per annum, the maximum accumulated stockpile capacity is unlikely to exceed 1 million tonnes, at an average blended grade of 0.074% MoS_2 (see Figure 16.13 below). The average stockpile grade decreases from 0.078% to 0.072%, as deferred marginal-grade material becomes more prominent.

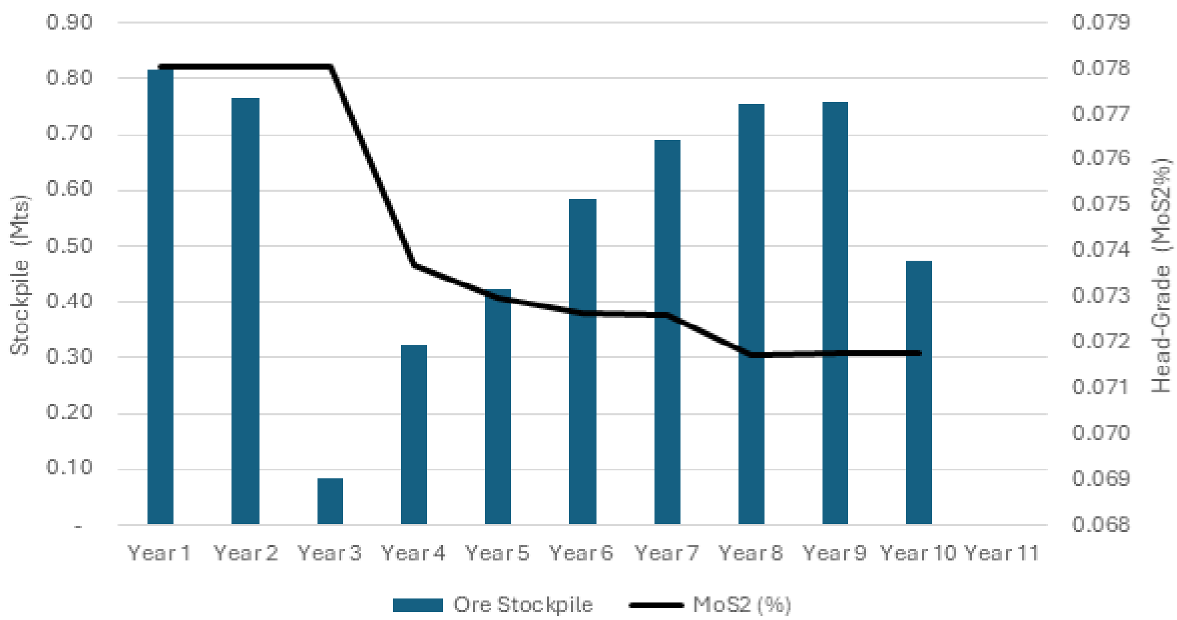


Figure 16.13 ***Ore Stockpile and Blending Schedule***
Source: *AMPL, 2025*

Figure 16.14 to Figure 16.18, below, show the targeted open-pit mining areas by year, with snapshots in grey for Years 1, 2, 3 and 5, and the final LOM pit (Year 11).

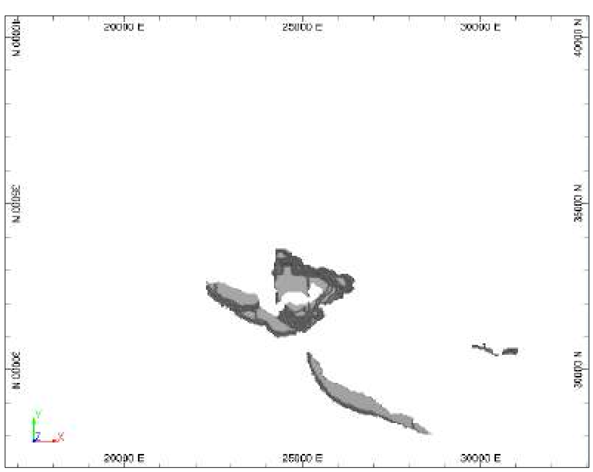


Figure 16.14 ***Year 1 LOM Open Pit***
Source: *AMPL, 2025*

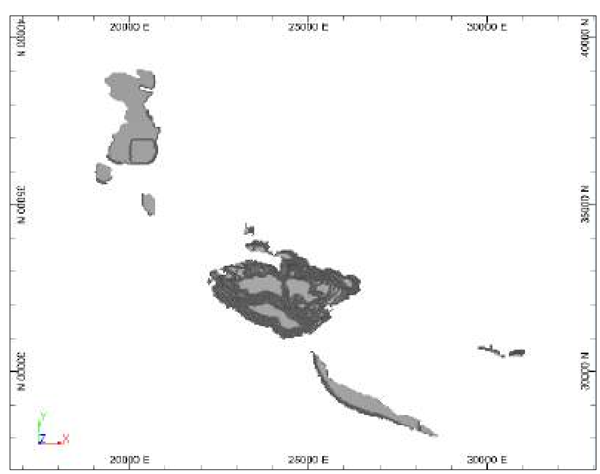


Figure 16.15 ***Year 2 LOM Open Pit***
Source: *AMPL, 2025*

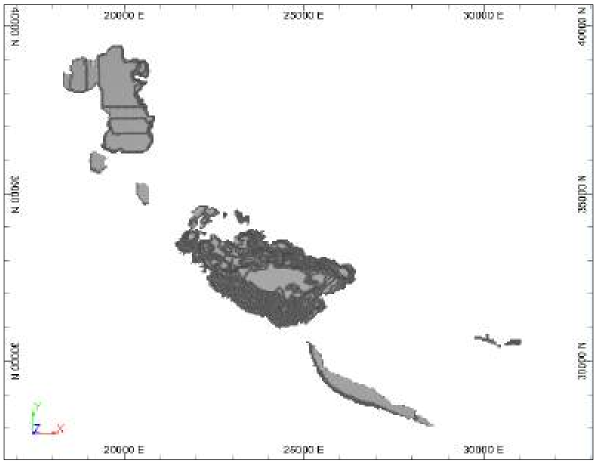


Figure 16.16 Year 3 LOM Open Pit
***Source:** AMPL, 2025*

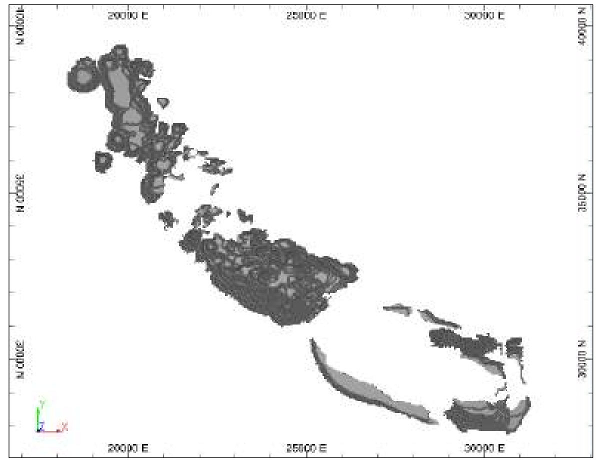


Figure 16.17 Year 5 LOM Open Pit
***Source:** AMPL, 2025*

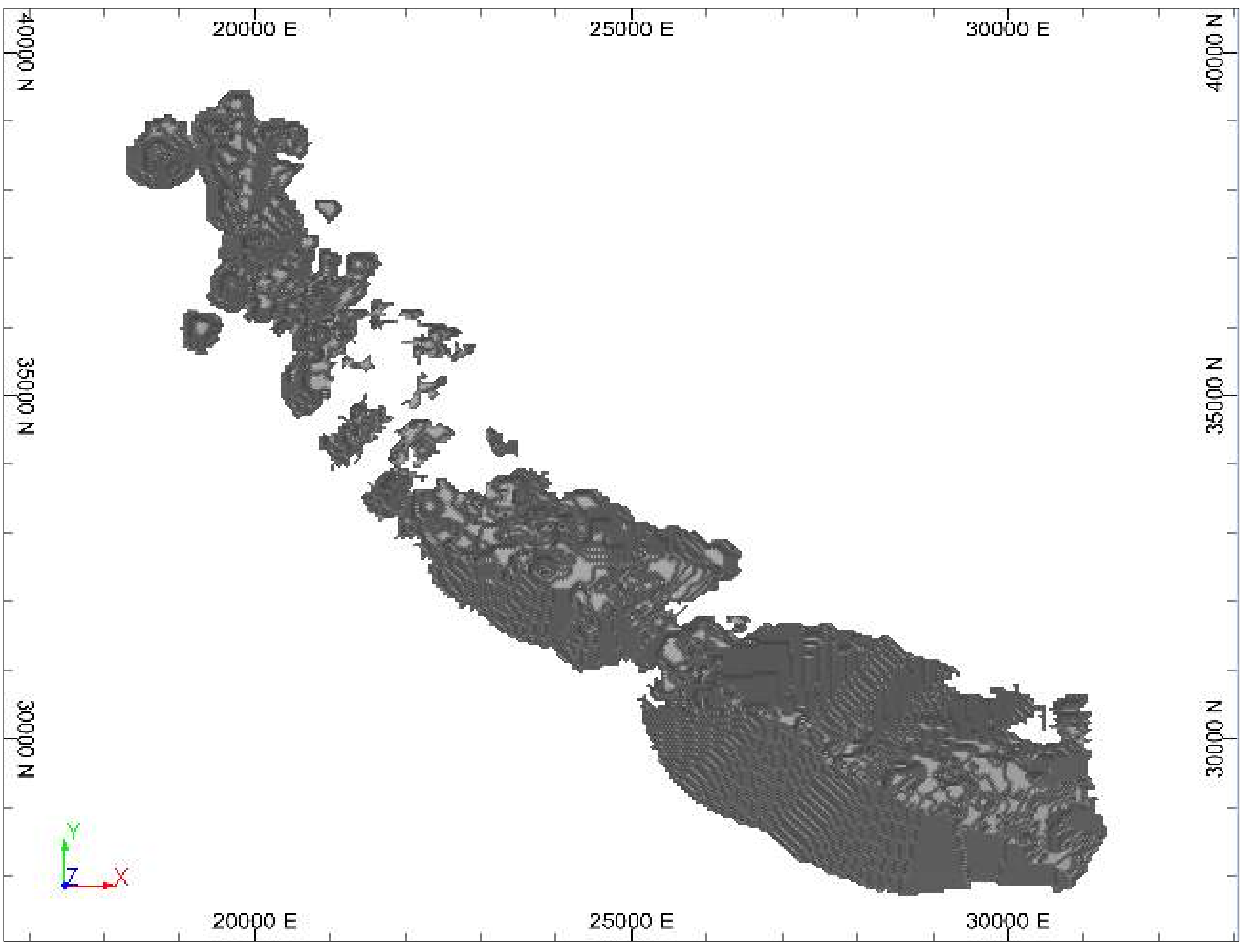


Figure 16.18 Year 10 LOM Open Pit
***Source:** AMPL, 2025*

16.7 PAG MANAGEMENT STRATEGY

According to the WSP-Golders report, the amended permit requires all PAG rock to be backfilled into a completed open pit and submerged under water. Consequently, different types of waste material must be tracked, with PAG and HR PAG stored separately to prevent any comingling.

The first pit scheduled for full extraction is Denak West in Year 6, which may need to be reserved for water storage, making it potentially available from Year 7 onwards. PAG material located near the pits will be bulldozed into a completed pit for permanent underwater storage. A temporary PAG stockpile will be positioned on the pit periphery, as close to the crest as possible, to minimise bulldozer push requirements. NAG waste will be placed in either the North or South-West WRSF, depending on haul distances from the active pit faces.

Strategically, prioritising the early completion of Denak West for use as a water and waste storage area may be feasible, subject to operational priorities and physical constraints. Further trade-off studies should be undertaken to confirm this approach. Prioritising Denak West's completion by Year 6 for storage from Year 7 is logical, if it aligns with the mine sequence, as Endako's pits (including Denak areas) are known to flood naturally, facilitating this setup and potentially lowering closure costs. However, this depends on geotechnical stability, water balance modelling and avoiding delays to ore extraction.

Subaqueous disposal is considered best available technology ("**BAT**") for PAG management, as endorsed by Canadian guidelines (*e.g.*, the Mine Environment Neutral Drainage ("**MEND**") programme). Isolating PAG underwater prevents sulphide oxidation, metal leaching and acidic run-off that could impact local waterways, such as the Endako River or Francois Lake. Separate storage of HR PAG adds an extra layer of precaution for higher-risk materials. This proactive approach is more sustainable than reactive treatment (*e.g.*, lime dosing), which can generate sludge and require energy-intensive operations.

16.8 MINING EQUIPMENT SELECTION

AMPL's estimates are partly based on previous work by WSP-Golder. Their model for calculating trucking requirements incorporates several key assumptions, including a site-wide maximum speed of 48 km/h, an in-pit maximum speed of 32 km/h, and a site-wide rolling resistance of 3%. The maximum access ramp grade was assumed to be 10%. Both mechanical availability and effective utilisation were assumed to be 100% in the modelling and were incorporated into the cost model. A combined truck queue, load, spot and dump time of approximately four minutes was applied. The vertical advance rate was limited to no more than four benches per period, with each bench being 13.4 m high.

The proposed equipment fleet comprises the following:

- **Production Fleet**
  - **Excavators (primary):** Four Komatsu P&H 2800 electric rope shovels
  - **Excavators (support):** Four Komatsu P&H 2100 excavators
  - **Haul trucks:** Fourteen Terrex MT4400 (218-tonne class trucks), and four CAT 793 Trucks (240-tonne class trucks)
  - **Production drills:** Four electric drills capable of drilling 31.8 cm (318 mm) diameter blast holes

- **Ancillary Fleet**
  - **Dozers:** Two CAT D10 dozers and two CAT D8 dozers
  - **Graders:** Two CAT 16M graders
  - **Integrated tool carrier:** One CAT 992 integrated tool carrier

Based on the above fleet, the mining capacity exceeds 46 to 48 million tonnes per annum. Average LOM ROM production is approximately 27 to 28 million tonnes per annum.

As part of care-and-maintenance activities, the mine mothballed certain previously used equipment, including three electric wire rope shovels and three production drills. These items would be overhauled before the mine restarts. A number of previously operated haul trucks also remain on site; these would be overhauled only if considered suitable and would then serve as backup units. All new haul trucks and support equipment would be purchased or leased. In 2021, equipment manufacturers assessed the existing fleet and provided Centerra with cost estimates for recommissioning usable items. AMPL has adjusted those estimates to reflect 2025 costs.

AMPL has not yet performed a detailed analysis of trucking requirements using haulage distances derived from the centroids of key mining areas, taking into account location, destination and scheduling period.

Developing a detailed, LOM estimate, based on actual truck-cycle times, would improve the accuracy of scheduled equipment utilisation, engine hours and overall fleet sizing. AMPL, therefore, recommends that this analysis be undertaken in future studies.

Figure 16.19, below, illustrates the forecast total major mining equipment fleet requirements for the Endako Mine over the LOM schedule. A full complement of production drill rigs and 240-tonne haul trucks would be in place by Year 2 to ensure sustained operational efficiency (see Table 16.8 and Table 16.9, below).

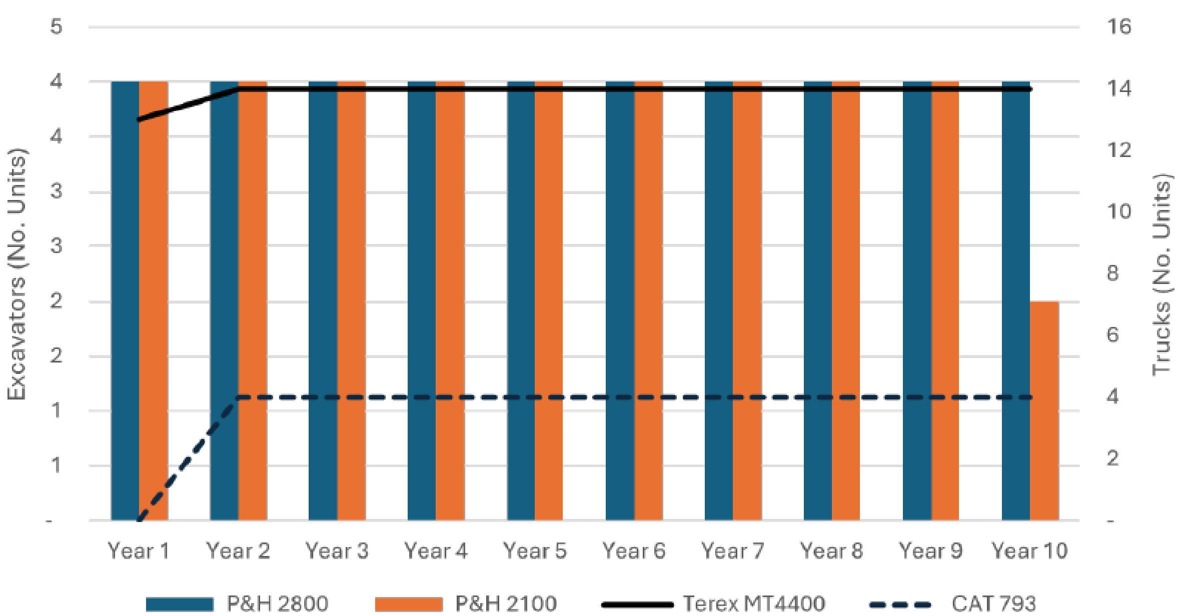


Figure 16.19 Forecast Mining Equipment Fleet Schedule (Trucks and Excavators)
***Source:** AMPL, 2025*

TABLE 16.8
EQUIPMENT FLEET SCHEDULE

Type	Model	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Excavator (primary)	P&H 2800	4	4	4	4	4	4	4	4	4	4
Excavator (support)	P&H 2100	4	4	4	4	4	4	4	4	4	2
Trucks	Terex MT4400	13	14	14	14	14	14	14	14	14	14
Trucks	CAT 793	-	4	4	4	4	4	4	4	4	4
Drills	Epiroc PV-271E	2	4	4	4	4	4	4	4	4	4
Grader	CAT 16M	2	2	2	2	2	2	2	2	2	2
Dozers (primary)	CAT D10	4	4	4	4	4	4	4	4	4	4
Dozers (backup)	CAT D8	4	4	4	4	4	4	4	4	4	4
Front-end Loaders	CAT 962	2	2	2	2	2	2	2	2	2	2
Front-end Loaders	CAT 993K	2	2	2	2	2	2	2	2	2	2

Source: *AMPL, 2025*

TABLE 16.9 FORECAST MINING EQUIPMENT FLEET		
Equipment	**Size**	**Quantity**
Hydraulic Shovel	29.8 m^3	4
Hydraulic Shovel	29.8 m^3	4
Rear Dump Truck	240	14
Grader	160 kw	4
Rotary Drill	38.1 cm	4
Grader	160 kW	3
Bulldozer	325 kW	4
Bulldozer	325 kW	4
Wheel Loader	WA900-8	2
Wheel Loader	WA900-8	2
Water Tanker	30,000 litre	1
Service Truck	20,400 gvw	9
Powder Buggy	6,800 gvw	11
Lighting Plant	10.1 kW	5
Pump	93.2 kW	4
Pick-up Truck	-	24
Total		**99**

16.9 MANPOWER

The Endako Mine operating labour is divided into two categories: operations and maintenance. A total of approximately 270 to 300 operations personnel, including both salaried and hourly workers, is estimated to be required for the mining operations.

17.0 RECOVERY METHODS

The Endako Mine has 2 existing processing facilities: an original mothballed processing facility (Old Plant), which had processing capacity of approximately 30,000 tonnes per day and a larger 52,000 tonnes per day processing facility on care and maintenance since 2015 (New Plant). Both processing facilities employed flotation recovery producing molybdenum concentrate.

Recovery of a molybdenum concentrate from mined ore will be achieved by reconfiguring and refurbishing the two existing processing plant facilities. The addition of new processing equipment is required for concentrate leaching, dewatering and drying. The combined processing plant will have a capacity to treat 75,000 tonnes per day or 27 million tonnes per annum. A summary process flowsheet of the Endako process plant is shown in Figure 17.1, below.

The existing in-pit primary crusher and a new second crusher, located close to the New Plant, will process the potentially economic ROM mineralisation.

The crushed potentially economic mineralisation will be ground in the SAG mill located in the New Plant followed by grinding in ball mills located in both the New Plant and the Old Plant.

The grinding products from the Old Plant and the New Plant will be processed in the New Plant flotation circuits. Concentrate will be produced by rougher/scavenger flotation, primary concentrate regrinding, first cleaner and scavenger flotation, secondary concentrate regrinding, secondary cleaner flotation and final concentrate thickening.

Final concentrate leaching, product dewatering and drying will be performed in a new facility constructed on the footprint of the existing ultra-pure plant (to be demolished).

The existing roaster complex would not be utilised in future operations. Molybdenum concentrate will be shipped to third party smelters.

17.1 PROCESS DESIGN CRITERIA

Table 17.1, below, summarises the key operating and design criteria for the major processing circuits based on the 2021 Hatch New Plant assessment and work by AMPL on New and Old Plants’ reconfigurations.

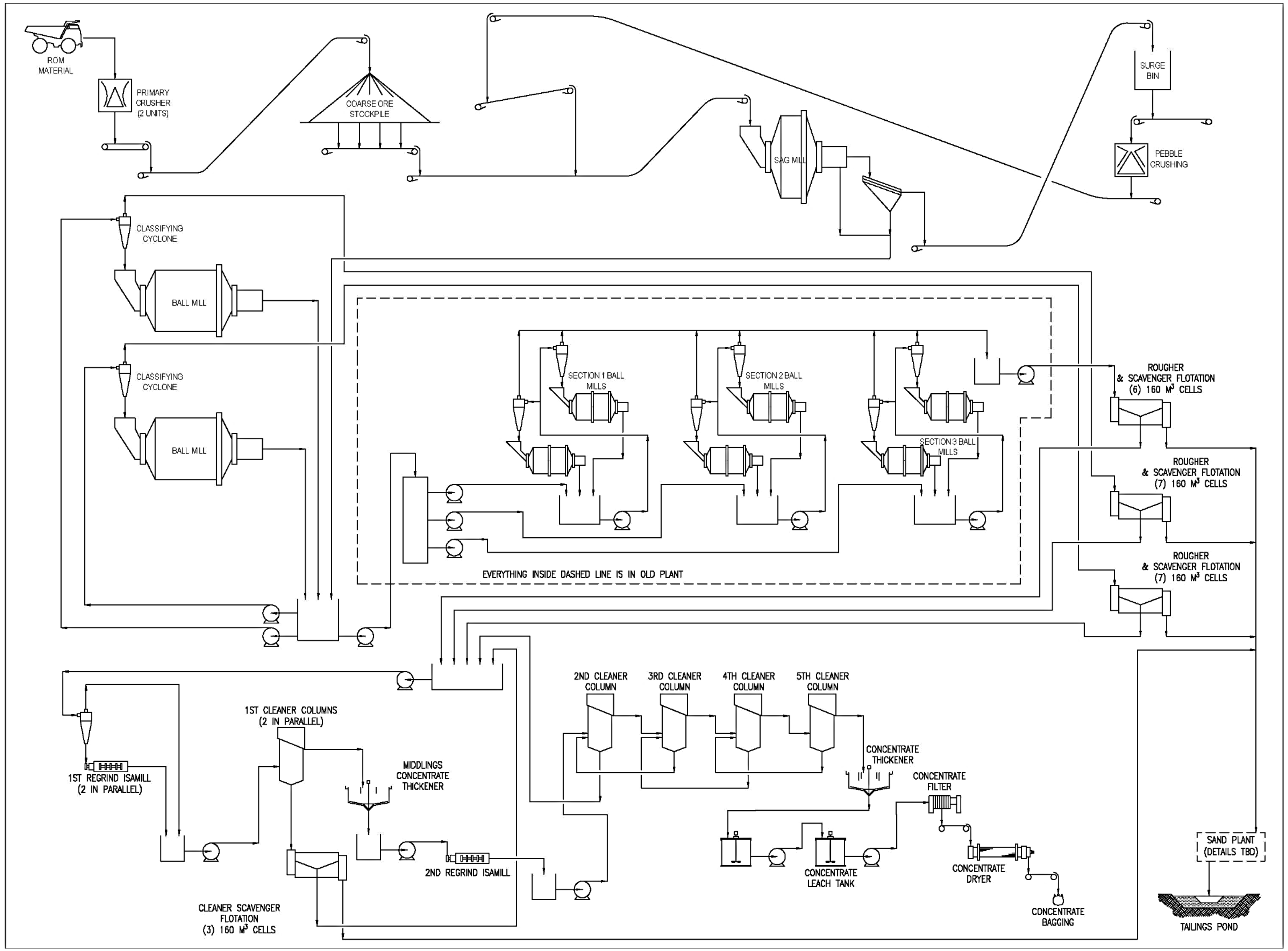


Figure 17.1 Processing Flowsheet

Source: *Concentrator Support Ltd., 2025*

TABLE 17.1
KEY PROCESS DESIGN CRITERIA

Area	Criteria	Unit	Nominal Value
General	Average Molybdenite (MoS_2) Head Grade, LOM	%	0.075
	Daily Throughput	t/d	73,553
	Process Plant Availability	%	92
	Overall Molybdenite Recovery	%	75.8
	Average Annual Molybdenite Production	t/yr	15,300
Crushing	Crushing Circuit Availability Target	%	75
	Crusher Throughput, Instantaneous (2 units combined)	t/hr	4,086
	Crusher Product (P80) Size	mm	113
Grinding	Bond Ball Mill Work Index (BWi)	kWh/t	14.9
	JK Axb Parameter (Inferred)	-	87.7
	Fresh Feed Rate, Instantaneous	t/hr	3,331
	Ball Mills Final Grind Size (P80)	µm	200
Flotation	Rougher-Scavenger Flotation Design Residence Time	min	21
	Final Concentrate Mass Pull	%	0.06
	Rougher Flotation Slurry Feed Flowrate, Total	m^3/hr	7469
	1st Cleaner Flotation Slurry Feed Flowrate, Total	m^3/hr	2386
	1st Cleaner Scavenger Slurry Feed Flowrate, Total	m^3/hr	2389
	2nd Cleaner Flotation Slurry Feed Flowrate, Total	m^3/hr	153
Concentrate Regrinding	1st Regrind Mill Product Size (P80)	µm	<55
	2nd Regrind Mill Product Size (P80)	µm	<16.4
Concentrate Thickening	Thickener Underflow Density	%w/w	50.0
Concentrate Dewatering	Filter Press Concentrate Cake Moisture Content	%w/w	12.0
	Concentrate Dryer Product Moisture Content	%w/w	3-5

Source: *Concentrator Support Ltd., 2025*

17.2 PROCESS DESCRIPTION

17.2.1 In-pit Gyratory Crusher

The presently installed gyratory crusher is a Kobelco 54-74 model unit with a 600 kW drive (see Figure 17.2, below). Final crushed product discharges onto a sacrificial conveyor. The existing in-pit crusher would be relocated to a new building outside the new ultimate open pit limits. A new concrete foundation and a pre-engineered building equipped with an access ramp would be constructed to house the existing primary crusher equipment. The existing rock breakers and electrical infrastructure would be relocated with the primary crusher. New piping for the hydraulic and lube unit will be installed. New electrical wiring and cable tray will be installed.

Figure 17.2 Existing to be Relocated – In Pit Crusher
Source: *AMPL, 2025*

To eliminate primary crushing as a process bottleneck, a second primary crusher (54-75) equipped with a 600 kW motor will feed a second overland conveyor. Product conveyed by the second overland conveying network will discharge onto the existing overland conveyor network to feed the coarse ore stockpile.

Potentially mineable resources from the open pit would feed either primary gyratory crusher system to produce a product with a target particle size (P80) of 113 mm. Mined potentially mineable Resources would be transported via haul truck to either primary crusher area where it is deposited in a dual-dump pocket above the primary crusher. The dump pockets would be designed with enough storage capacity to accommodate a truck load of potentially mineable Resources. A rock breaker would be installed at each dump pocket for use in breaking any oversized material before feeding into the crusher. Access roads and ramps for haul trucks for potentially mineable Resources delivery would be configured for each location of the crushers.

17.2.2 Coarse Potentially Economic Mineralisation Stockpiling

The coarse potentially economic mineralisation stockpile has a total design capacity of 290,000 tonnes, representing a maximum of about 4 days of operation at the original design throughput of 2,355 tonnes per hour (tph). Maximum design live capacity was about 32 hours of operation or 104,000 tonnes. At the higher planned throughput of 3,331 tph, the maximum design pile capacity would be sufficient for about 3.6 days total, with about 31 hours being live. One of the reasons to specify a second in-pit crusher was to reduce or eliminate mill downtime caused by any primary crusher outage exceeding the design 31-hour live capacity of the coarse-ore stockpile at the higher planned throughput.

Stockpiled potentially mineable Resources will be reclaimed using four apron feeders, located in the reclaim tunnel beneath the stockpile and discharges onto the SAG mill feed conveyor feeding the SAG mill in the New Plant building.

17.2.3 Grinding

The revised grinding circuit for Endako will consist of a SAG mill and two main ball mills located in the New Plant, plus 6 of the 10 smaller mills located in the Old Plant configured as ball mills. The New Plant and Old Plant ball mills will operate essentially in parallel, with all mills operating in closed circuit with hydrocyclones. The grinding circuit will be designed to treat 3,331 tph of fresh feed and grind it to a target flotation feed size F_{80} of 200 µm.

The SAG mill installed at Endako is 10.97 m in diameter and 5.41 m in length with a total installed power of 12,750 kW. The SAG mill is driven by two 6,375 kW motors in a dual pinion drive arrangement. Ore inside the SAG mill discharges first through a 70 mm aperture grate and then trough a trommel with an aperture of about 19 mm. SAG trommel oversize is sent to the pebble crushing circuit which consists of wash screen and a 274 kW cone crusher. Crushed pebbles return to the SAG mill feed while SAG trommel undersize advances to a common discharge pump box for the SAG and two main ball mills.

The common discharge pump box will feed each main cyclone-feed pump plus a transfer pump to bleed a portion of the combined flow to the Old Plant to reduce the burden on the main ball mills.

Each main ball mill is fed by the underflow of a dedicated cluster of 840 mm diameter cyclones, while cyclone overflow slurry gravitates to a dedicated rougher and scavenger flotation bank. The two main ball mills installed in the New Plant measure 6.7 m in diameter and 10.36 m in length, each with 8,250 kW of installed power. Each mill is driven by two 4,125 kW motors in a dual pinion drive arrangement. The target grind size (P80) of the cyclone overflow is 200 µm and is analysed using a sampler and particle size monitor.

The portion of combined SAG mill and main ball mill discharge that will be pumped to the Old Plant will be discharged to a centralized distribution pump box (possibly the Scavenger Feed pump box for those familiar with the plant). To meter the flow from this box to each of the three grinding sections, a trio of variable speed pumps will be used – one feeding each section. Because these section feed pumps will be required to operate at very low heads, wear rates are expected to be low enough to obviate the need for standby pumps. For each section, both mills will discharge into a common cyclone feed pump box equipped with a pair of variable-speed pumps (duty and standby). The pumps will discharge into a single pipe, with the discharge split to feed each cyclone. While each grinding section previously was comprised of one rod mill and one ball mill, the rod mills will be reconfigured such that they exactly match the ball mills. All mills to be used in the Old Plant are 3.8 m diameter × 4.6 m long and are equipped with 1120 kW motors. Cyclones will be single vertically mounted 800 mm units feeding each mill. Note that this layout was chosen to reflect very limited available space at the feed end of the mills in the Old Plant.

Cyclone overflow from the three Old Plant sections will be combined and pumped back to a dedicated flotation bank in the New Plant.

The use of the grinding capacity of Sections 1, 2 and 3 in the Old Plant is estimated to add enough extra grinding capacity (6,714 kW total installed for the 6 mills) to allow grinding of the higher throughput.

Process water addition to the mills and cyclone feed pump boxes will be controlled to maintain the target slurry densities in the mills and cyclone feed to achieve consistent grind size and slurry density feeding the rougher flotation circuit.

Collector (diesel oil) and frother (pine oil) will be added to the grinding circuit in established Endako practice to condition the flotation slurry ahead of the rougher flotation circuit.

17.2.4 Rougher Flotation

The New Plant rougher flotation circuit initially consisted of two parallel seven-cell banks of 160 m^3 cells giving 24 minutes of installed residence time. Each bank was (and will be) fed by the cyclone overflow of its main ball mill cyclone overflow. Of the seven cells, the first three cells were called roughers while the remainder were called scavenger cells despite the fact all concentrates advanced to cleaning. To accommodate the planned extra mill throughput, a row of six 160 m^3 cells will be added to the New Plant to parallel the existing two rows. This new row will treat the combined cyclone overflow from the Old Plant. Tailings from the rougher-scavenger banks will be pumped to the tailings management facility. Rougher and scavenger concentrates will flow by gravity from launders into a pump box common to the three parallel flotation banks.

For the present study, the installation of additional rougher flotation capacity is the only change envisioned for the New Plant flotation process.

17.2.5 Regrinding and Cleaner Flotation

Prior to regrinding and cleaner flotation, rougher and scavenger concentrate is passed through a cyclone. Cyclone sands are ground in open-circuit in a pair of parallel M1000 IsaMills with 503 kW motors. The design target Isamill product size is a P80 of 41 µm. The Isamill product and cyclone fines (55 µm P80 according to the Hatch design) are recombined and pumped to a pair of parallel first cleaner columns measuring 4.88 m in diameter and 14.5 m in height.

First cleaner concentrate reports to an intermediate thickener (Middlings Concentrate Thickener), while first cleaner tails reports to a bank of (3) 160 m^3 cleaner scavenger flotation cells. Concentrate from the cleaner scavenger cells returns to first regrind cyclone feed, while the cleaner scavenger tailings report to final tails.

Thickened concentrate slurry from the Middlings thickener is passed through a single second-stage regrind mill (M1000 IsaMill with 503 kW motor). The design target product P80 size of the second mill is finer than 16.4 µm.

The product from the second regrind mill is pumped to a group of four cleaner columns measuring 2.44 m in diameter and 12.0 m in height. The columns are operated as second, third, fourth and fifth-stage cleaners and are operated in counter-current mode; concentrate from each stage is cleaned again in the following stage, while the tailings of each stage return to the preceding unit. The final concentrate from the fifth cleaner stage is pumped to the final concentrate thickener while the second cleaner tails return to the first regrind cyclone feed.

The final concentrate thickener is 9 m in diameter and is designed to settle the concentrate to a slurry solids density of 50% by weight. Thickener underflow will be pumped to an agitated concentrate stock tank in a new facility for molybdenum leaching, dewatering, drying and packaging.

17.2.6 Concentrate Product Preparation Facility

The concentrate packaging facility will receive concentrate at 50% solids by weight from the concentrate thickener underflow. The concentrate will be leached in hydrochloric acid in an agitated tank then filtered in a new concentrate filter press. Cake discharged from the filter press will be dried to between 3% and

5% moisture in a new paddle dryer and then packaged. The circuit will be erected in a new building adjacent to the New Plant.

An isometric representation of the new concentrate product presentation facility is presented in Figure 17.3, below. The configuration and location of the circuit and building is presented in Figure 17.4, below.

Figure 17.3 Concentrate Dewatering Circuit
Source: *Hatch, 2022*

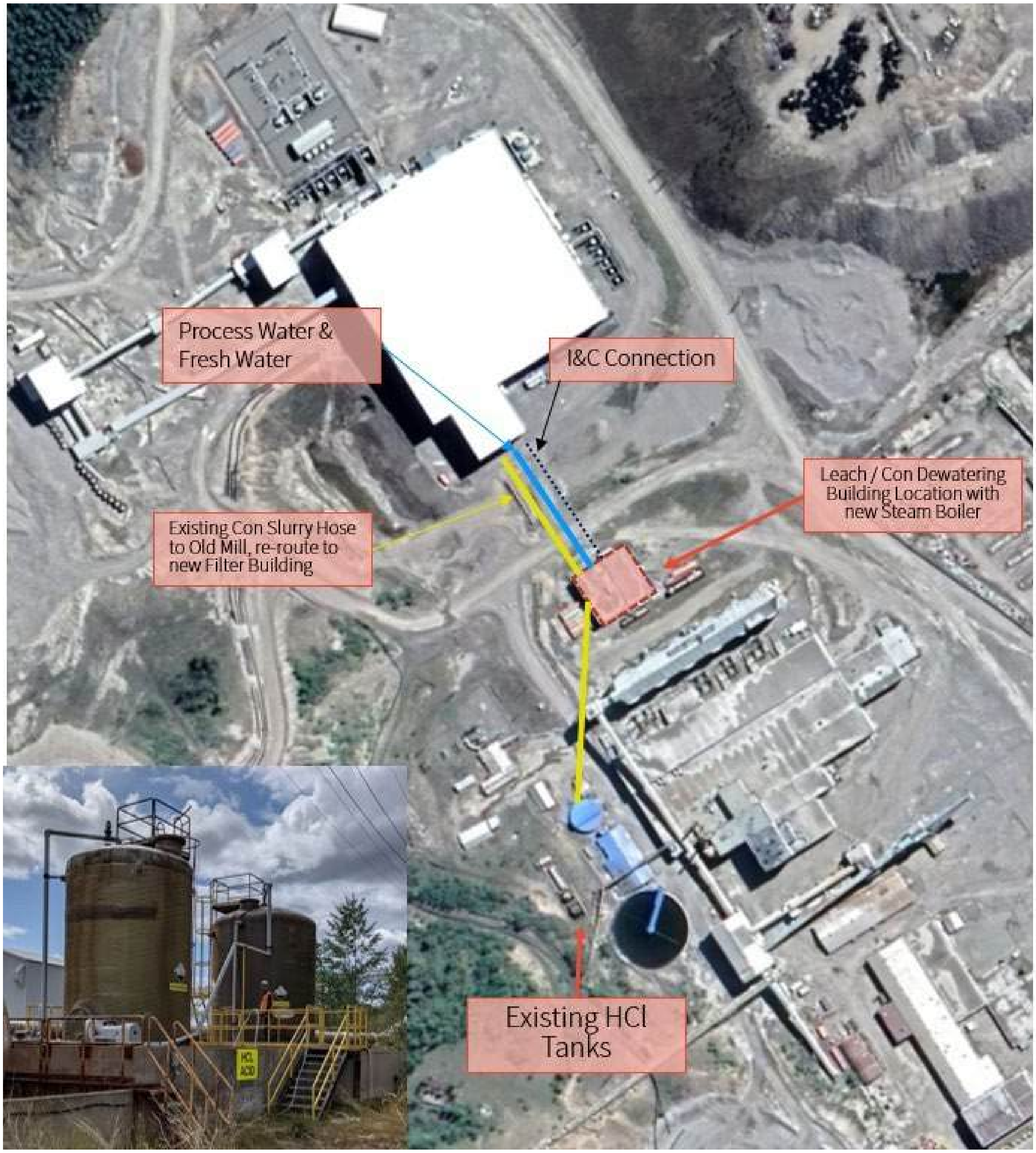


Figure 17.4 Proposed Modified Site Plan
***Source:** Hatch, 2022*

17.3 REAGENTS

17.3.1 Diesel Fuel Oil

Diesel fuel is used in the Endako process as a molybdenum flotation collector. The liquid is transported to the site in bulk tankers and off-loaded into the diesel storage tank through a transfer pump. The diesel fuel reagent is metered to the ball mills and flotation cells using the plant reagent system.

17.3.2 Pine Oil

The frother used in the Endako flotation process is pine oil. This chemical is transported to the site in bulk tankers and off-loaded into the pine oil storage tank through a transfer pump. Pine oil reagent is metered to the ball mills and flotation cells using the plant reagent system.

17.3.3 Sodium Cyanide

Sodium cyanide is used in the Endako flotation process as a depressant for copper minerals. The reagent arrives on site in briquette form in 1-tonne bulk bags and is dissolved in fresh water. It is metered into the flotation process using the plant reagent system.

17.3.4 Lime (Hydrated)

Hydrated lime is used in the Endako process to depress iron and copper sulphide flotation. It is shipped to the site in bulk tankers and off-loaded into the storage silo on site. The Lime is slurried in fresh water then metered to the process using the plant reagent system.

17.3.5 Flocculant

To improve settling rates in the Concentrate Middlings Thickener and Final Concentrate Thickener, Flocculant (shipped in 1-tonne bags) is dissolved in fresh water and metered into the thickener feed streams.

17.3.6 Hydrochloric Acid

For use in concentrate leaching prior to filtration, hydrochloric acid is brought to the site in bulk tankers and pumped into the fiber-reinforced plastic storage tanks.

17.3.7 Plant Water Supply

- **Plant Process Water** is comprised of water reclaimed from the tailings pond and water overflow from the Middlings and Final Concentrate Thickeners. This water is used for grinding process water and flotation circuit dilution water.

- **Fresh Water** is pumped from the nearby Francois Lake and used for pump gland-seal water, reagent preparation water and process make-up. The freshwater pumping system consists of a pump station located at Francois Lake followed by a booster pumping station.

17.3.8 Plant Services

In general, the plant services (such as blower air, compressed air, fire water, etc.) required for the restart of the Endako process plant are all existing and are designed for the operation of the facilities for the original nameplate production of 52,000 tonnes per day. In their 2021 report, Hatch indicated that the throughput record for the New Plant was 88,094 tonnes milled in a day; because of this, only minimal changes to the required capacities of existing utilities at the site are expected. Recommissioning of the plant services will involve servicing of existing equipment and/or replacement of damaged and inoperable equipment and wear parts as identified by Hatch in 2021.

17.4 PROCESS PLANT REFURBISHMENT REQUIREMENTS

Refurbishment requirements for the present study use the study Hatch produced in 2021 as the baseline. To the Hatch study, AMPL has added items relevant to expansion of the plant throughput to the approximately 75,000 tonnes per day envisioned.

Additions to the Hatch scope of refurbishment (see Appendix) include the following:

- Installation of a second gyratory crusher and overland conveying system to supplement the capacity of existing in-pit crusher.
- Modification of New Plant mill discharge pump box and laying of pipeline to transfer a portion of mill discharge slurry to the Old Plant.
- Installation of a pump and piping system within the Old Plant to distribute slurry between grinding Sections One, Two and Three.
- Installation of new cyclone feed pumps and cyclones for the three Old Plant grinding sections.
- Reconfiguration of the rod mill in each Old Plant grinding section to exactly match the corresponding ball mills (each section used one rod mill and one ball mill of the same size).
- Installation of an Old Plant cyclone overflow transfer system to return finished slurry to the New Plant.
- Installation of additional rougher flotation capacity in the New Plant to handle the increased production rate.

The New Plant refurbishing plan and cost estimates, as provided by Hatch, were used and costs increased by factoring to reflect equivalent 2025 Dollars. Pricing for equipment for the expansion scope was based on scaling of prices from other projects and from vendor budgetary quotes where possible. Pricing for reconfiguration of the mills in the Old Plant was estimated based on discussion with a mechanical contractor and vendor component pricing (essentially drive pinions and mill liners).

17.5 TAILINGS NEUTRALIZING AND DEWATERING FACILITIES

Neutralisation of tailings produced in the processing plant, containing cyanided thiocyanates, will be performed using the alkaline chlorination process. In this process, the cyanides and thiocyanates are completely oxidised, the solution rendered non-toxic and metals of cyanide complexes (copper, zinc) precipitated from the liquid, in the form of hydroxide compounds.

After detoxification, tailings slurry will be pumped from the processing plant to the tailings dewatering facility located at the south side of TP 1. Pipelines will be installed around TP 1-3 to convey coarse, fine and cyclone bypass tailings to TP 1-3. The tailings transport pipelines will consist of high-density polyethylene (HDPE) pipes and spigots that will allow control of the tailings beach and deposition of coarse tailings into the paddock cells.

The return water pipeline from TP 1-3 to the process plant will consist of an upgraded barge with a pumping system to convey impoundment water in a HDPE pipe to the process plant.

17.6 MANPOWER

The process plant operating labour is divided into two categories: Mill Operations and Mill Maintenance. A total of 118 personnel, including both salaried and hourly workers, is estimated to be required for the process plant.

18.0 INFRASTRUCTURE

As the Endako Mine was a past producing mine infrastructure for the Project is existing, though refurbishing and upgrading of facilities is required.

18.1 EXISTING SITE INFRASTRUCTURE

The infrastructure at the Endako Mine site includes the following:

- Access roads, BC power grid power supply to the site, nearby railway line and fresh water supply network;
- Tailings management facility with 2 disposal areas: TP-1 and TP-3;
- Reclaim water ponds;
- Administration, warehouse, change house, laboratory, mine shops and first aid station;
- 1 reclaimed tailings disposal area – TP-2; and
- 2 non-operational roasters (which will not be re-activated).

A general plan for site infrastructure is presented in Figure 18.1, below1. Refurbishing and upgrading of all facilities will be required and is included in the restart plan and cost estimates. A more detailed presentation of scope for these facilities is described in this section for:

- Electrical Power;
- Water Supply;
- Tailings Management Facility;
- Water Management and Treatment;
- Site Communications;
- Truck Shop;
- Maintenance Shop;
- General and Administration and Mine Dry Building; and
- Assay Laboratory.

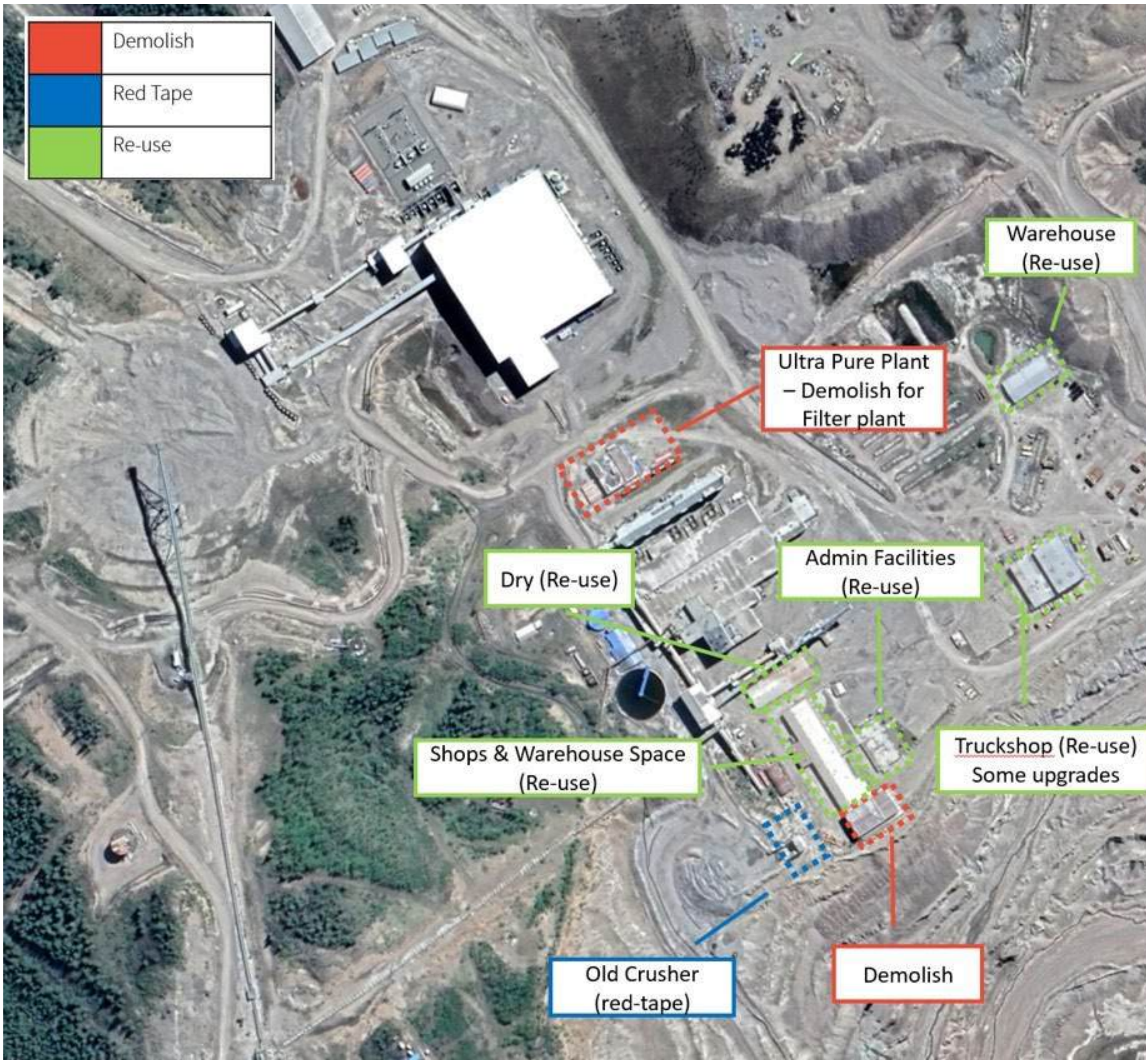


Figure 18.1 Site Layout for Endako Mill Expansion
Source: *Hatch, 2022*

18.2 SITE POWER

The Endako Mine site is grid connected to the BC Hydro grid (Figure 18.2, below) through the Glenannan (GLN) sub-station by means of an 8.5 km 69 kV overhead line that was upgraded during 2010.

Figure 18.2 BC Power Grid Map
Source: *Hatch*

As part of the 2010 upgrades, the GLN sub-station also underwent upgrades to fully support the new 69 kV, 84 MW main sub-station at the Endako Site.

The condition of the line as well as the equipment within the main sub-station on the site have been maintained and continuously energised while the rest of the plant was put on care and maintenance. It was determined that one of the breakers in the main sub-station H-configuration requires repair. Repair of the breaker is critical as the present main sub-station configuration offers redundancy and without it, the plant would have no spare power path in the event of a failure of this breaker.Due to this, it was recommended that the breaker be repaired.

A new mine 69 kV sub-station transformer would replace the existing sub-station transformer that is near its end of life and not usable. Cables and electrical equipment/infrastructure (*i.e.*, 69 kV sub-station transformer, MCC, E-House) were included to provide power to the new open pit dewatering loads. A buried 69 kV feed coming to this sub-station would supply power to the new sub-station, as well as provide a point of distribution to the open pits.

18.3 TAILINGS AND STORAGE FACILITIES

The Endako Mine is a molybdenum mine operation located approximately 20 km southwest of the town of Fraser Lake, British Columbia, Canada. The Endako Mine began operations in 1965. Tailings generated from processing ore were placed in tailings ponds that include Tailings Ponds 1, 2 and 3 (TP-1, TP-2 and TP-3). Operations were suspended in December 2014 and the site entered into care and maintenance in 2015. The tailings management facilities (TMF) are identified in the general arrangement shown in Figure 18.3, below; the existing quantity of tailings in the historical facilities is unknown.

Figure 18.3 Existing TP-1, TP-2 and TP-3 Facility General Arrangement
Source: *Ausenco, 2025*

The Endako Mine Restart Project seeks to expand the existing TP-1 and TP-3, placing tailings within the existing facilities, and makes use of existing infrastructure to the extent practicable. The existing dam crest of TP-1 and TP-3 are at elevations 997 metres above sea level (m.a.s.l.) and 982 m.a.s.l., respectively. The primary design objectives of the expansion are to secure confinement of tailings and the protection of regional groundwater and surface water during mine operations and after closure. The following have been considered in the design of the TP expansion and water management facilities:

- Maximizing the use of existing TPs and existing tailings management infrastructure to minimize environmental disturbance.
- Staged development of the facility over the LOM.
- Control, collection, and removal of water from the facility during operations for recycling as process water to the maximum practical extent.

The proposed tailings management for the restart is to expand TP-1 and TP-3 to contain an additional 258 Mt of new tailings over a 10 year period (see Figure 18.4, below). All tailings will be pumped by pipeline from the process plant to a hydrocyclone facility, with cyclone underflow, overflow and bypass pumped to the TPs.

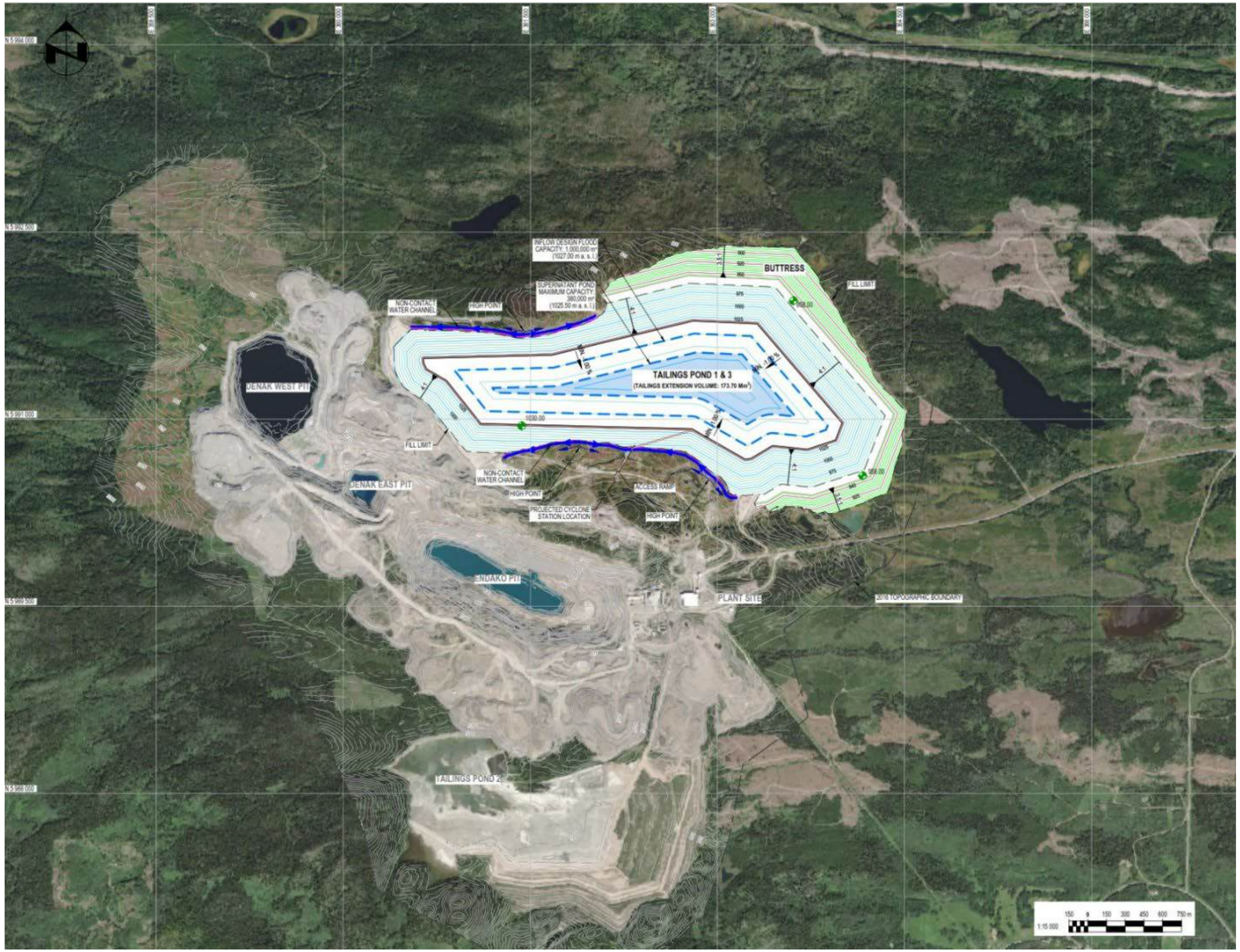


Figure 18.4 Proposed Tailings Pond General Arrangement

Source: *Ausenco, 2025*

18.3.1 Existing TP-1 and TP-3

The tailings ponds were developed using the upstream construction method. Tailings were placed at approximately 35% solids through spigots around the facility's perimeter. Tailings were passively segregated as they flowed down the interior tailings beach, with the coarse material settling near the discharge points. The coarse tailings from the beach were excavated to construct a berm for discharging tailings and creating the next lift (see Figure 18.5, below).

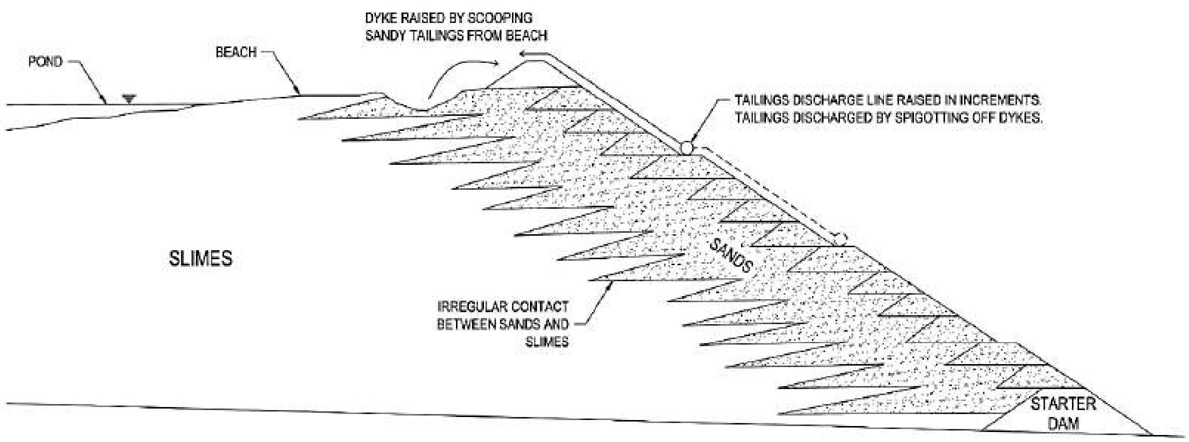


Figure 18.5 Historical Endako TP Construction Method
Source: *Ausenco, 2025*

The Engineer of Record (EoR) for the Tailings Ponds was Golder, then WSP, who acquired Golder, until 2024. In 2025, WSP resigned from the EoR position and Knight Piesold Consulting (KP) took over the EoR responsibilities.

18.3.2 Tailings Pond 1 (TP-1)

The initial storage facility, TP-1, was built in 1964, and deposition into this facility began in 1965 and continued until 2014. Currently, the facility's dam crest is approximately 997 m.a.s.l. The dam hazard classification for TP-1 is "Extreme" according to Canadian Dam Association guidelines (2019) since there is no spillway during operations and the facility needs to manage the Probable Maximum Precipitation (PMP).

18.3.3 Tailings Pond 3 (TP-3)

TP-3 is situated east of TP-1 (see Figure 18.3) and was commissioned around 2012. This facility was constructed to expand the overall storage capacity of the Endako Mine and to mitigate potential stability issues associated with increasing the height of TP-1. TP-3 is located at the rear of the TP-1 watershed and its crest elevation is 974 m.a.s.l. The dam hazard classification for TP-3 is "High" according to Canadian Dam Association guidelines (2019) since it is buttressed by TP-1.

18.3.4 Topography and Drainage

The Endako Mine is located on the Neckako Plateau at an approximate elevation of 914 m.a.s.l. bounded on the by the Francois Valley and to the north Endako River Valley. The surficial geology of the area is mapped as a 1 m to 3 m thick layer of glacial till overlying bedrock. The topography of the Endako Mine Site is generally characterised by topographic relief in the form of gullies near creeks. The majority of the TP-1 and TP-3 were constructed over natural basin-shaped topography low that drains to the east to northeast.

18.3.5 Facility Design: Expansion of TP-1 and TP-3

A trade off study between filtered tailings disposal and cyclone sand was performed to determine which disposal technology to utilise to provide secure containment of the tailings. Based on the trade off study, cycloned tailings was chosen to create the tailings facility embankments while depositing both fine tailings and whole tailings into the facility. The design for the Endako Mine Restart utilises the existing TMFs TP-1 and TP-3 to contain tailings in a merged facility named TP 1-3. The design of TP 1-3 and associated water management facilities has taken into account the following:

- Staged development of the facility over the life of the Project;
- Flexibility to accommodate operational variability in tailings (plant shutdowns, deposit variability, and placement during variable climate conditions); and
- Control, collection and removal of contract water from the facility during operations for reuse as process water to the maximum practical extent.

The design criteria for TP 1-3 considered the following requirements for cyclone tailings:

- Tailings slurry storage requirement: approximately 258 Mt (178 million m^3);
- Average tailings density: dry density of 1.45 t/m^3;
- Limiting watershed disturbance to a single catchment basin;
- Inflow design flood (IDF) is to contain the PMP; and
- Design earthquake is for the Maximum Credible Earthquake (MCE).

TP 1-3 includes the following embankments:

- “Ring” embankment.

Tailings will be transported from the mill to a hydrocyclone system by a pipeline. At the hydrocyclone, tailings will be separated into a coarse and fine fraction. Coarse tailings will be utilised to construct TP 1-3 embankments, and the fine tailings will be discharged into the facility. When coarse tailings are not required for construction of embankments, the tailings will bypass the hydrocyclone and be discharge directly into the facility.

During the development of TP 1-3, forested areas will be cleared and grubbed and topsoil removed for later closure.

A buttress located at the base of TP 1-3 is planned to provide additional stability to the existing and planned embankments of TP 1-3 based on stability analyses. The buttress will be constructed from waste rock.

The TP 1-3 embankments will be constructed from coarse tailings generated from the hydrocyclone underflow. The coarse tailings will be transported to the upstream tailings beaches for deposition and dewatering and paddock cells for passive dewatering by gravity to ensure water from the tailings remains within TP 1-3. Dewatered coarse tailings will be rehandled for use as construction sand to construct the tailings embankments using excavators, haul trucks, dozers and compactors. Fine tailings and the cyclone bypass tailings will be transported to deposition points within TP 1-3, which does not interfere with dewatering of the coarse tailings.

TP 1-3 will have a maximum height of 1030 m.a.s.l. (see Figure 18.6, below).

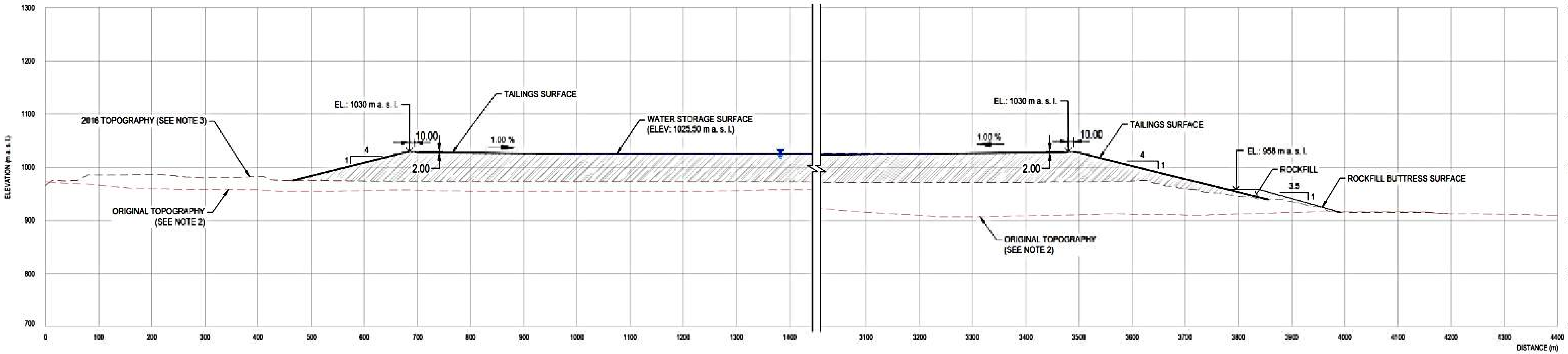


Figure 18.6 Typical Section of TP 1-3
***Source:** Ausenco, 2025*

18.3.6 Tailings Management Facility Stability

Stability analysis of the planned TP 1-3 was completed incorporating historically deposited tailings material properties along with the foundation. A section through the highest portion of the embankment was selected as the critical section for slope stability analysis. Stability was assessed using the limit-equilibrium modelling software Slide2™, a module of the Rocscience™ 2025 software suite. Analyses were undertaken for both static and pseudo-static (earthquake loading) conditions with the calculated factors of safety (FOS) higher than the minimum required values of the CDA guidelines. The tailings embankment is designed to withstand potential dynamic displacement without release of tailings during the maximum design earthquake event.

18.3.7 Tailings Deposition and Return Water

Pumps at the process plant will convey slurry tailings to the hydrocyclone located on the south side of TP 1-3. Pipelines will be installed around TP 1-3 to convey coarse, fine and cyclone bypass tailings to TP 1-3. The tailings transport pipelines will consist of high-density polyethylene (HDPE) pipes and spigots that will allow control of the tailings beach and deposition of coarse tailings into the paddock cells.

The return water pipeline from TP 1-3 to the process plant will consist of an upgraded barge with a pumping system to convey impoundment water in a HDPE pipe to the process plant.

18.3.8 TP 1-3 Water Management

All run-off generated by precipitation, which falls on TP 1-3, is considered contact water. Contact water will be collected and utilised for process water or retained prior to being treated (if required) and released to the environment utilised for process water. Primary components of the contact water management system include the following:

- Interior of TP 1-3; and
- Exterior Slopes of TP 1-3.

The TP is designed to contain the 24-hour PMP with a minimum freeboard of 2 m. No water from other facilities is allowed to be pumped to the tailings facility. Water that reports to the tailings ponds from tailings disposal operations will be reclaimed for process water. A floating water reclaim barge will be located on the south side of the facility. Non-contact water, above TP 1-3, will be conveyed around the facility to natural drainages or existing diversion channels. Figure 18.7, below, shows the proposed water management infrastructure.

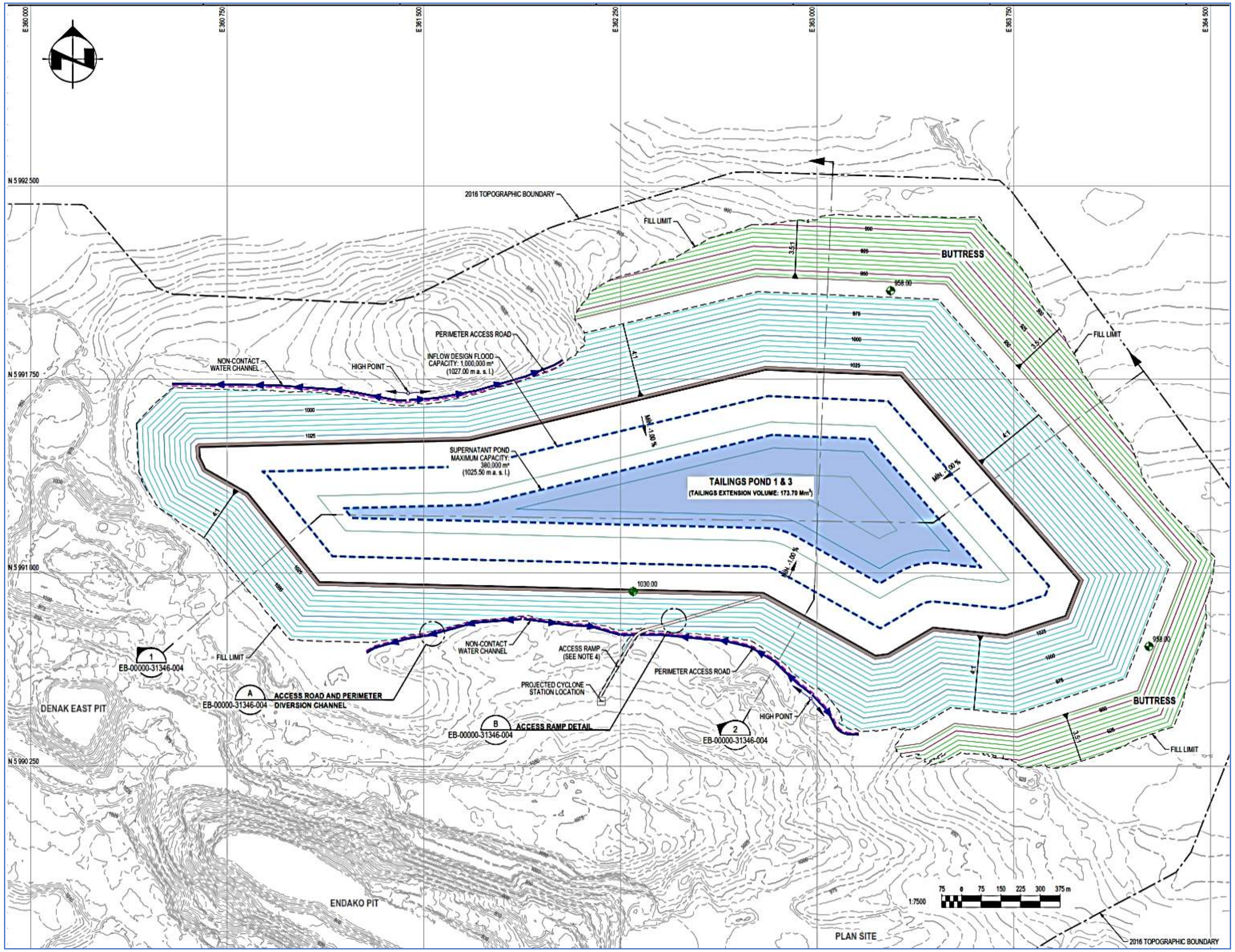


Figure 18.7 TP 1-3 Water Management Infrastructure
Source: *Ausenco, 2025*

The existing seepage control measures will be utilized in the restart operations to collect superficial seepage and treat (if required) and released to the environment.

18.3.9 TP 1-3 Monitoring

To support construction-level design and permitting, a detailed geotechnical monitoring plan will be prepared that defines the roles and responsibilities of key stakeholders (Owner, Operator, Engineer) for safe and stable TP 1-3 construction and operation. Monitoring will be accomplished through both measurements of monitoring points (*e.g.*, survey monuments, piezometer and inclinometer readings) and visual observations of surface conditions. Monitoring will incorporate existing instrumentation and monitoring infrastructure (*e.g.*, piezometers, survey monuments) where possible.

18.3.10 TP 1-3 Closure

The general closure design strategy includes placing a topsoil and vegetative cover to stabilise the tailings surface. Growth medium stripped during construction of TP 1-3 will be stockpiled for future placement over the waste rock surface and on the exposed embankment surfaces during reclamation. In addition, a spillway will be constructed to ensure freeboard criteria is maintained post closure, similar to TP-2.

Downstream slopes of the TP 1-3 embankments have been designed with 4H:1V slopes that are sufficiently flat for effective re-vegetation. For this Study, a 30 cm-thick topsoil cover or growth medium layer above the covered tailings and downstream embankment slopes was considered. The closure cover will be graded with drainage swales to convey surface run-off to the closure spillway. Surface water will be conveyed and discharged into natural drainages. Maintenance may be required to provide repairs for any damage created by larger or more intense storms.

18.4 SITE WATER MANAGEMENT

The site wide water strategy has been developed based on the proposed mine arrangement and will be implemented during the initial mine development phase and be adjusted as necessary throughout the mine operations and closure phases.

Water management considers two types of water: contact and non-contact water. Contact water is water (surface or ground) that has been exposed to excavated materials (*e.g.*, mineralised materials, tailings and waste rock) or process facilities (*e.g.*, water within the process plant circuit). "Non-contact water" is diverted around the mine facilities and otherwise does not interact with the mine.

The water management strategy aims to:

- Intercept and divert surface water run-off that naturally drains toward the mine facilities (*i.e.*, non-contact water) to minimise the generation of contact water that will require management and potentially treatment prior to discharge.

- Intercepting run-off from waste rock storage facilities to either use in the process or potentially treatment prior to discharge.

- Ensure a continuous supply of makeup water to the process plant.

- Provide erosion and sediment controls for surface water run-off from mine facilities during mine operations to reduce sediment loading to water management system or sediment release to the environment.

18.4.1 Meteorological Data and Design Criteria

The criteria used for the design of the water management facilities are summarized in Table 18.1, below. The closest climate station to the Project Site, with a sufficient long-term data set, was the Fraser Lake North Shore Station, summarised in Table 18.2 and Table 18.3, below.

TABLE 18.1
WATER MANAGEMENT DESIGN CRITERIA

Item	Design Criteria
Meteorological Data	Environment Canada Climate Normals for Fraser Lake North Shore Station (109C0LF)
	IDF curves from Environment Canada (Burns Lake; 1091169)
	Evaporation data from historical restart water balance report (Pereira, SRK, 2022)
Collection Ditches	Hydraulic design (*i.e.*, ditch size, erosion protection) for a 1/100-year, 24-hour storm event
Collection Ponds or Sumps	Ponds sized to contain run-off from a 1/100-year, 24-hour storm event
Non-contact Diversions	Hydraulic design (*i.e.*, ditch size, erosion protection) for a 1/100-year, 24-hour storm event

TABLE 18.2
FRASER LAKE NORTH SHORE CLIMATE NORMALS (1981-2010)

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Rainfall (mm)	5.4	4.3	5.4	16.6	40.9	57	57.4	46.5	47.9	46.2	19.3	3.9	**350.7**
Snowfall (cm)	42.5	25.2	20.3	6.5	1.1	0	0	0	0	8.7	29.5	41.3	**175.1**
Precipitation (mm)	47.9	29.5	25.7	23.1	42	57	57.4	46.5	47.9	54.8	48.8	45.2	**525.8**
Snow Depth at Month-end (cm)	42	41	21	0	0	0	0	0	0	2	12	30	N/A

TABLE 18.3
DESIGN STORM EVENTS DATA

Storm Duration (1/100 Year Return Period)	Precipitation (mm)	Precipitation Intensity (mm/hour)
5 minutes	13.23	158.8
10 minutes	17.9	107.4
15 minutes	24.3	97.1
30 minutes	28.8	57.4
1 hour	28.8	28.7
2 hours	31.5	15.7
6 hours	33.89	5.7
12 hours	377.2	3.1
24 hours	49.5	2.1

18.4.2 Water Management Structures

This section summarises a list of proposed water management structures for the Endako Mine Site. The major structures are as follows:

- **Diversions** – Diversion ditches are required to divert non-contact runoff away from the facilities and to minimize the amount of contact runoff to be collected and managed. The design criterion for the diversion ditches was the conveyance of 1:100-year peak flow without overflow.

  In additional to internal non-contact diversions, a diversion along the northwest extent of the WRSF will capture and convey existing unclassified streams flow around the facility north to Watkins Creek.

- **Collection Ditches** – Collection ditches collect contact runoff from the waste rock dump and process plant area. Contact water collection ditches are designed to be trapezoidal in section with 2H:1V side slopes, bottom width of 1 m and minimum required depths ranging from approximately 0.5 to 1 m. Ditches will be excavated into the existing overburden and/or bedrock or formed by grading surface material to achieve the required ditch geometry. Ditches constructed in overburden will require erosion protection along their base and side slopes. Erosion protection will consist of riprap overlying non-woven geotextile. Collection channels will be constructed around the perimeter of the WRSF to capture and convey runoff to the adjacent ponds or sumps.

- **Collection Sumps** – Where topography does not allow gravity drainage back to an existing/proposed pond, an excavated sump will be required. Sumps have been placed in topographic low points with collection ditched reporting to them. Water accumulated in the sumps will be pumped for use in the process or sent to treatment, if required.

  Ditches and water management facilities were sized using estimated peak flow rates and flood volumes from the rational method and frequency analysis results.

18.4.3 Site-Wide Water Balance

A preliminary site-wide water balance analysis was performed for the Project. In this analysis, a comparison between water requirements and available water from the collection system was made to identify the site-wide water balance. This analysis has been made for the Site's average climate conditions. The following water components were considered in this calculation:

- Surface run-off from precipitation on WRSF, process plant area and pits;
- Evaporation from ponds and pits; and
- Water sent to the TSF and water reclaimed.

It is understood that the process plant will require 34 million m^3 of water per year and that the expected recycling rate is around 60%, requiring approximately 13.6 million m^3 of make-up water that will be needed on a yearly basis. Based on the annual water balance, there is approximately 12 million m^3 of water available from Site run-off. At this stage, it is assumed that the water quality of the Site run-off will be of sufficient quality to use in the process plant (see Table 18.4, below).

TABLE 18.4
SITE-WIDE WATER BALANCE – AVERAGE CONDITION (M^3/MONTH)

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Surface Run-off													
Net Run-off Collected in Pond 1, 2 and 3[1]	248,474	153,027	133,315	116,361	199,877	274,501	275,756	223,431	238,845	281,760	253,142	234,468	**2,632,956**
Precipitation on Pits	287,477	177,047	154,241	131,337	214,189	297,505	298,179	241,640	267,206	323,610	292,878	271,272	**2,956,582**
Groundwater in Pits[2]	18,600	16,800	18,600	18,000	18,600	18,000	18,600	18,600	18,000	18,600	18,000	18,600	**219,000**
Water Collected in Sumps	33,173	20,430	17,799	15,998	29,087	39,475	39,753	32,204	33,173	37,952	33,797	31,303	**364,144**
Tailings Water													
Water in Tailings	6,688,877	6,041,566	6,688,877	6,473,106	6,688,877	6,473,106	6,688,877	6,688,877	6,473,106	6,688,877	6,473,106	6,688,877	**78,756,128**
Reclaimed Water	5,284,854	4,773,416	5,284,854	5,114,374	5,284,854	5,114,374	5,284,854	5,284,854	5,114,374	5,284,854	5,114,374	5,284,854	**62,224,889**
Available Free Water[3]	468,008	422,717	468,008	452,911	468,008	452,911	468,008	468,008	452,911	468,008	452,911	468,008	**5,510,413**
Available Water													
Total Available Water	**1,055,731**	**790,021**	**791,962**	**734,607**	**929,760**	**1,082,392**	**1,100,295**	**983,882**	**1,010,135**	**1,129,930**	**1,050,728**	**1,023,651**	**11,683,094**

Note:
[1]After evaporation
[2]Based on a 25 m^3/hr steady state inflow
[3]Assumes ⅓ of the remaining water is available – accounts for seepage, entrained water.

18.4.4 Groundwater

Previous water balance evaluations for the Site (Pereira, SRK, 2022) indicated overall contributions from groundwater to the site-water balance would be negligible, but did not include a formal estimate. To confirm this assumption, Ausenco calculated preliminary high-level estimates of groundwater inflow, which may contribute to the overall site-wide water balance. Based on these initial estimates, groundwater contributions to the water balance are expected to be relatively low compared to other sources. The average monthly and corresponding annual volume of water anticipated to enter the pit is based on a steady-state flux of 25 m^3 per hour (approximately 110 USGPM), resulting in an annual inflow of 219,000 m^3.

The estimated inflow was derived using both site-specific data as well as literature values. A model domain and steady state simulation was defined and evaluated using SEEP/W in Seequent's Geostudio™ 2D. The model dimensions were based on the current estimates of the proposed 10-year pit dimensions, incorporating a longitudinal cross-sectional profile of the pit, as well as the surface perimeter of the pit. The model domain was defined as a unit thickness of 1 m.

No reliable data from geotechnical or hydrogeological investigations of the site were available; thus, the model used an estimated hydraulic conductivity ($1 \times 10E^{-7}$ m/s) based on literature values for fractured bedrock representing the quartz-monzonite surrounding the pit. The water table was estimated to be 10 m below ground surface surrounding the pit, and distal constant head boundaries were used to evaluate steady-state flow conditions. For simplicity, bedrock was considered to be isotropic and homogeneous. Unconsolidated surficial materials noted in bore hole logs around the Site (alluvium and till) were considered negligible contributors to the overall flow, as these materials constituted only a thin layer several meters thick overlying the bedrock.

18.4.5 Water Treatment

A water treatment plant will be constructed to treat water from initial pit dewatering, and to treat, as required, seepage that requires discharge to the environment. It is anticipated that the mineral processing plant will recycle seepage during operations.

A water treatment plant will be required for the Project. SRK completed the design basis for the water treatment plant (Pereira, SRK, 2022). The design will be similar to the Kemess water treatment plant, excluding the selenium treatment component. The plant will process:

- Average of 1,250 m^3 /hr of contaminated water;
- Expected influent concentration of total molybdenum during operations of 5 milligrams per litre (mg/L) on average and 10 mg/L at maximum; and
- Discharge limit for total molybdenum in the treated effluent of 0.1 mg/L.

18.4.6 Water Treatment Overview

The process flow diagram for the proposed water treatment plant is presented in Figure 18.6, below. The proposed treatment consists of metals co-precipitation with ferric hydroxides followed by clarification, neutralisation and filtration steps. Water from pit dewatering and seepage collection (if required) will be fed to the water treatment plant. Influent water will be pumped to a mechanically agitated tank where ferric chloride ($FeCl_3$) solution will be added at a Fe:Mo ratio of 10:1 (Aube & Stroiazzo, 2000). Ferric chloride precipitates and forms ferric hydroxide $Fe(OH)_3$, which adsorbs molybdate under acidic conditions. The pH will be decreased to target levels by adding hydrochloric acid (HCl).

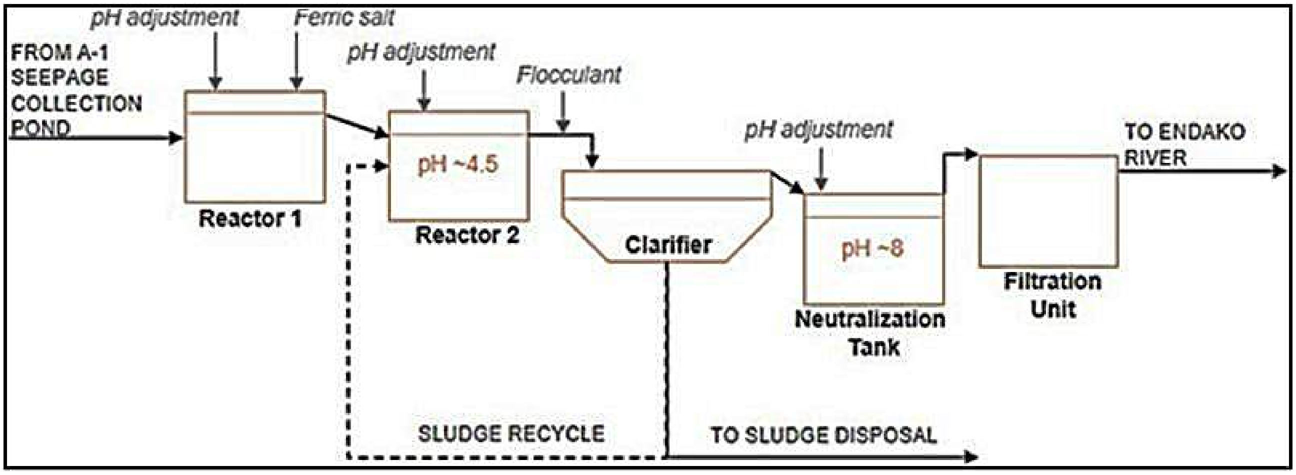


Figure 18.8 Process Flow Diagram of Proposed Water Treatment Plant
Source: *SRK Consulting (Canada) Ltd., 2022*

Water containing ferric hydroxide precipitates will then overflow to a second mechanically agitated tank where the pH can also be adjusted.

Overflow from the ferric iron reactor will then flow to the clarifier. A flocculant with low aquatic toxicity will be added to the clarifier. The addition of the flocculant will result in the formation of larger particles and will enhance the settling of the precipitates.

The clarifier sludge will mostly consist of ferric hydroxide with some adsorbed molybdenum. A portion of the sludge will be recycled to the reactors to act as seed to form the precipitates. Sludge production rate is expected to be 10,000 m^3/year (20% solids by weight). The sludge will be trucked to the tailings facility for co-disposal with the tailings.

Water from the clarifier will overflow to a filtration unit to remove any remaining particles that do not settle in the clarifier. Following the overflow from the filtration unit, pH will be adjusted to 8 by the addition of sodium hydroxide (NaOH) solution in a mechanically agitated neutralisation tank.

The treated effluent will be discharged to the Endako River discharge point, if concentrations meet discharge limits. If treated water does not meet the end-of-pipe targets, it will be recycled to the water treatment plant feed or to the contingency discharge line to TP 1-3. The effectiveness of this treatment should be assessed with bench scale test work.

The treatment system should include the following:

- Flocculant storage, preparation, and dosing system;
- Agitated reactor tanks;
- Filtration unit;
- Clarifier;
- Clarifier underflow recycle and waste pump systems;
- Clarifier overflow tank for discharge of treated water;

- Flush water and dilution water pump systems;
- Sump systems for the reactor, clarifier, and reagent preparation areas;
- Instrumentation and controls; and
- Instrument air system.

18.5 SITE COMMUNICATIONS

Business Local Area Network/Wireless Area Network (LAN/WAN) provide intranet/internet services to the Endako Site. There is fibre optic cabling available to all buildings that services the Site's business network including the phone system. The phone system, which utilises Voice over Internet Protocol (VoIP) technology, provides voice services for the Endako Mine Site.

The VoIP system is used for routine and emergency telephone and fax communications within the Endako Mine Site and between the Endako Mine Site and the Public Switched Telephone Network (PSTN) through a satellite system.

The Ethernet base control system network, with redundant communication switches, is separated from the business network by the firewall. The control system network services the equipment that directly controls the process equipment. There is fibre optic cabling available for long distance communication to remote locations. Wireless hubs are used to communicate with wireless devices.

All systems underwent upgrades and were expanded in 2010 to accommodate additional locations at the Endako Mine Site. Mobile phone service is available at the site.

18.6 SURFACE INFRASTRUCTURE

18.6.1 Truck Maintenance Shop

The truck maintenance shop is a pre-engineered steel building designed with two different heights.

The higher roofed portion of the building accommodates 4 maintenance bays for 240-tonne haul trucks, 2 of them located on the east side of the building and 2 of them located on the west side of the building. The west side of the building has been modified with one large door for easier access by the large haul trucks into the building. The same modification is planned for the east side of the building. There is a 35t/5t overhead crane covering the 4 maintenance bays.

The lower roofed portion of the building is being used for servicing the light vehicles and various maintenance activities. Costs for general repair and cleaning have been included to restore general conditions for future use.

The truck maintenance shop has existing electrical power supply, with no action required (see Figure 18.9, below).

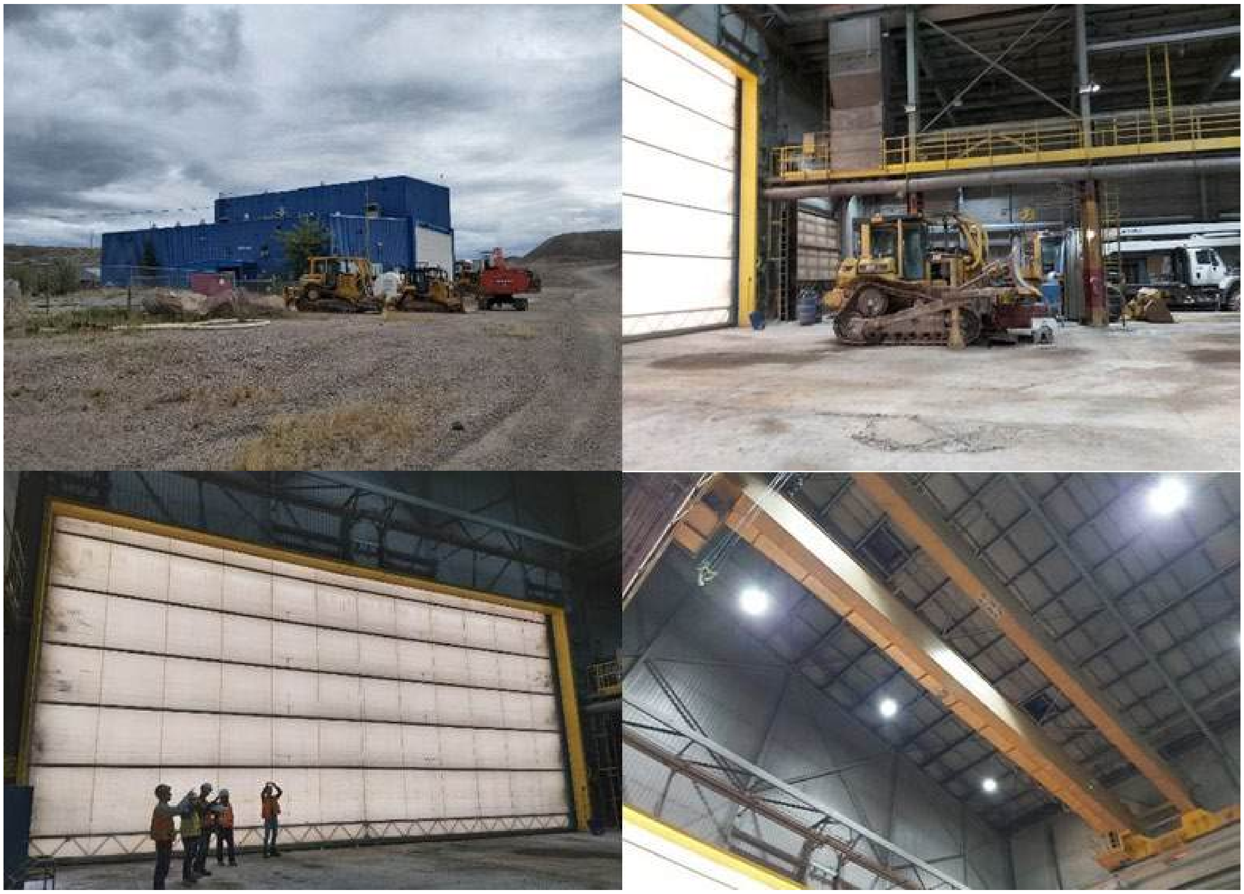


Figure 18.9 Truck Maintenance Shop
Source: *Hatch*

18.6.2 Maintenance Building

The maintenance building (see Figure 18.10, below) is a pre-engineered steel building located on the west side of the plant next to the open pit. It includes the following areas:

- Shops;
- Warehouse;
- Tire Bay;
- Machine Shop;
- Machine Shop Annex;
- Administration;
- Storage; and
- Oil Storage.

The inside of the building requires minor refurbishing and general maintenance to ready it for future use. There is a 2.5 cm wide crack on the slab on-grade isolation joint suggesting that foundations on the taller part of the building, close to the pit, are sliding in the pit direction. Hatch recommends removing the overhead crane inside and demolishing this part of the building. Power supply must also be re-established to this building.

Figure 18.10 Old Maintenance Shop
Source: *Hatch*

18.6.3 Administration Office Building

The administration office building is located on the south side of the maintenance building and is a pre-engineered, single story steel building (see Figure 18.11, below).

Figure 18.11 Administration Office Building
Source: *AMPL, 2025*

Cleaning will be required to restore the office for use. Power connection must be restored to the existing building, as there is no longer any power. As power is required for the administration office building early in the construction schedule, a 4160 V feed was carried to be run from the mill building, as well as a dry-type 4160/600 V transformer and short 600 V cable to provide power to the administrative building during construction.

18.6.4 Assay Lab

The present assay laboratory is housed in a single-story, wood building, which is beyond its serviceable life as a laboratory (see Figure 18.12 and Figure 18.13, below). A new, pre-engineered steel building will be constructed to house the assay laboratory. Where required, existing laboratory equipment will be refurbished or upgraded to today's standards.

Figure 18.12 Existing Assay Laboratory
Source: *AMPL, 2025*

Figure 18.13 Assay Laboratory Equipment
Source: *AMPL, 2025*

18.6.5 Warehouse Buildings

There are two warehouse buildings in good condition and in use by the processing plant. One located close to the guard house is a fabric building with concrete foundations and consists of open shelving for small to large items. The second building, located close to the truck maintenance shop, is a pre-engineered steel building with an insulated roof and wall cladding. These are routinely maintained, as shown in Figure 18.14, below.

Figure 18.14 Warehouse
Source: *Hatch*

18.7 MANPOWER

The surface department includes non-mine and processing plant maintenance personnel.

G&A staff include senior management and other non-operations (*e.g.*, Accounting, Human Resource, Health and Safety, etc.) services for the Endako Mine.

Administration is comprised of senior and general management, accounting, engineering, geology, environmental and purchasing. As well as direct salaries and fringe benefits for personnel, other components include employee re-location, travel allowances for business away from the property, insurance (property, business interruption, and political risk), taxes, permits, and licenses, mining rights fees, professional fees and operating surface vehicles for the personnel.

Accounting functions would include providing payroll, accounts payable and receivable processing, expenditures budgeting and forecasting and other corporate cost accounting for the entire operation and reporting to the head office.

The environmental components costs would be associated with monitoring of the Endako Mine's environmental performance and reclamation work.

Procurement encompasses all functions associated with on- and off-site procurement of materials and supplies, warehousing and inventorying, transportation from point of origin to the site, and other associated support services. Actual freight costs for items required by the Endako Mine, processing plant and maintenance departments are included in those department's costs. The main cost components comprise wages and warehouse supplies. Salaries and fringe benefits for staff and warehouse supplies, such as personal protective gear (gloves, etc.) and small equipment (pallet lifters, forklifts, etc.) parts for operating the warehousing, purchasing and logistics groups, are included. Surface support includes unloading and loading of trailers and shipping containers, movement of materials on-site and maintenance of the warehousing and associated facilities.

Human resources encompass all functions associated with personnel and industrial relations, health and safety, training and community relations. Personnel and industrial relations costs comprise salaries and fringe benefits for the staff to undertake recruitment of required personnel, managing of company salary policies and fringe benefits, managing and negotiating of collective agreements with hourly employees and overseeing of company policies and procedures. Health and safety provide for salaries, fringe benefits and supplies for the services of on-site first aid personnel and all first aid supplies and vehicles required by them.

Community relations costs will provide funds to aid in supporting local community efforts and facilities.

Security would be provided on a contract basis by a security firm and include all personnel and supplies. Security surveillance equipment would be provided by the Endako Mine to the security firm. Small security equipment for the security personnel (metal detectors, etc.) would be provided by the contractor. Security personnel will also supply on-site emergency medical services.

The operation is and will be managed by a senior management team led by the General Manager.

All supervision, employees and contractors report to or through these positions.

The surface department and G&A staff would be comprised of approximately 151 personnel.

19.0 MARKET STUDIES AND CONTRACTS

The Endako Mine will produce a molybdenum concentrate that will be sold to a smelter(s) for further processing to metal. The smelters will charge a toll processing fee per tonne of concentrate.

Smelter payment prices are based on paying mines for molybdenum metal contained in MoO_2 concentrate minus smelter charges. This PEA has used the 3-year trailing average price for October 2022 to end of October 2025, published by Metal Platts, for the long-term price to be paid by smelters. Metal Platts is the primary recognised source for prices for molybdenum concentrate sales pricing worldwide.

The long-term MoO_2 price, based on the 3-year trailing average price included in this study, is US$49.73/kg (US$22.50/lb).

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Endako Mine holds permits for the current status of the project in care and maintenance. There are no compliance issues at the Endako Mine with permits or environmental regulations.

- Robust weekly, monthly and quarterly surface and groundwater monitoring programs to satisfy permit requirements;
- Full suite of Management Plans covering wildlife, soil, vegetation, sedimentation and erosion, fugitive dust, metal leaching and acid rock drainage (ML/ARD) monitoring;
- No regulatory fines on record; and
- Ongoing discussions with Nadleh Whut'en and Stellat'en First Nation on an Impact Benefit Agreement are nearing completion.

Tailings:

- Quarterly monitoring of instrumentation by site staff. and
- Yearly dam safety inspections by engineer on record (EOR), with a third-party Dam Safety Review was completed on March 18, 2025 in conjunction with the *BC Ministry of Energy, Mines and Low Carbon Innovation.*

Water use at Endako is limited during care and maintenance. However, water quality, both on-site and off-site, is carefully monitored to ensure compliance at locations specified in the permits. Also, detailed aquatic studies in the surrounding environment are conducted at sites in care and maintenance to ensure no adverse impacts are present.

20.1 WATER MANAGEMENT, WASTE MANAGEMENT AND MONITORING

20.1.1 Overall Water Management Strategy

The restart proposes a significant shift from the current passive care and maintenance model to an active water treatment strategy. The core of the new plan is to stop relying solely on natural dilution or passive settling and instead build dedicated infrastructure to minimise water use, maximise recycling and chemically treat contact water before it enters the environment.

Water withdrawal, use and discharge operate within a Continuous Improvement Framework while the Endako Mine remains in care and maintenance. The restart of operations will require amendments to the primary provincial permits and water licenses, necessitating updates to the operational plan and the aquatic effects monitoring program.

20.1.2 Key Water Management Techniques

The primary water management strategy centres around segregation and treatment. Non-contact water is diverted away from the Endako Mine works, water is recycled directly in the mill and the contaminated water stream is captured and directed to a water treatment plant prior to discharge. The improvements to water management centre around three main pillars: active water treatment, cyclone sand tailings and infrastructure reconfiguration. Collectively, these will achieve a high-water recovery target and manage effluent.

20.1.3 Infrastructure Improvements

The restart includes additional pumping capacity to manage the flow between the Old Plant and New Plant circuits, ensuring that mill process water is recycled internally to minimise the volume that needs treatment. The concentrate dewatering within the plant and tailings dewatering also maximise recycling. These recycling technologies reduce fresh make-up water demand and the volume of water requiring treatment.

New pipelines will be constructed to transport water from the existing TP-1 to the new water treatment plant.

20.1.4 Tailings and Dewatering

Pumps at the process plant will convey slurry tailings to the hydrocyclone located on the south side of TP 1-3. Pipelines will be installed around TP 1-3 to convey coarse, fine and cyclone bypass tailings to TP 1-3. The tailings transport pipelines will consist of HDPE pipes and spigots that will allow control of the tailings beach and deposition of coarse tailings into the paddock cells.

The return water pipeline from TP 1-3 to the process plant will consist of an upgraded barge with a pumping system to convey impoundment water in a HDPE pipe to the process plant.

The plan for the use of cycloned sand tailings allows for steeper, more stable dam walls, increasing the capacity of the existing footprint to store water and tailings without expanding into new watersheds. The use of cycloned sand for engineering steeper dam designs is granted by explicit regulatory approval after robust engineering justification.

20.1.5 Water Treatment Facility

A dedicated active water treatment plant will be constructed to manage and mitigate the site's key water concerns and contaminants: total suspended solids (TSS), molybdenum and sulphate and meet discharge criteria set under the CleanBC Industry Fund ("**CIF**") utilised at the Site. The treated water will be discharged in a controlled, permitted manner via a pipe directly into the Endako River. The treatment sludge will be safely disposed of in a designated disposal cell within the TMF to prevent the contaminants from re-dissolving and re-entering the environment.

20.1.6 Permits

The primary permits for operations require amending from care and maintenance status to operational status. As the Endako Mine restart does not contemplate a land disturbance of more than 50% of the previously permitted area, it does not trigger an environmental assessment.

20.1.6.1 Mines Act Permit

The amendment must address the full mine plan and a full application will be required.

20.1.6.2 Environmental Management Act Permits

- **Waste Water Discharge** – The current authorisation is valid only for care and maintenance and must be amended before restart.

- **Refuse Discharge Permit and Air Discharge Permit** are also currently scaled for care and maintenance and require amendments.

20.1.6.3 Water Sustainability Act Licenses

The license to withdraw from Francois Lake remains valid. Still, it may require verification of compliance with the Water Sustainability Act and potentially updated terms to align with modern environmental flow restrictions.

The water treatment plant must meet the stringent requirements of the CIF. The CIF sets short-term (2022-2026) and medium-term (2027-2031) water quality targets for molybdenum in the Endako River. The water treatment plant's ability to achieve a reliable concentration of less than 1.0 mg/L for molybdenum in discharge water from the water treatment plant is the most critical for obtaining the necessary permit amendments for the restart. The discharge to the Endako River is also subject to Yinka Dene 'Uza'hné (Water Law), which is the self-governance policy developed by the Nadleh Whut'en and Stellat'en First Nations, which states the goal for the Endako River is to meet water quality guidelines for aquatic life or ensure no increase over background levels.

The permitting process for the Endako Mine restart will require treating Indigenous approval as a fundamental prerequisite rather than a parallel consultation step. Integration of processes to obtain consensus on restart plans, closure plans, water chemistry and geochemistry will require communication and facilitation of technical oversight.

20.2 CLOSURE PLAN

20.2.1 Acid Rock Drainage/Metal Leaching ("ARD"/"ML")

Monitoring at the site indicated that no acid rock drainage has been detected in tailings seepages, waste rock seepages, or pit water to date. It was concluded that ARD was not occurring and was not expected due to the high neutralizing potential of the non-potentially acid-generating material. Despite these general, site level neutral conditions some isolated areas pose a potential long-term risk. Both areas are assumed to become acidic at the same time.

- **South Dumps** – Identified as generating acidic drainage that will drain to the pit; and

- **Pit Walls** – There are PAG zones containing pyrite and magnetite in the south walls of the Endako pit and East Denk pit. The areas are above the final water elevation.

It is recognised that there are difficulties with applying standard ARD/ML modelling to molybdenum mines. Molybdenum ores often have a combination of minerals that both generate and neutralise acidity when exposed to water. Additionally, the chemical reaction involved in ARD/ML release from molybdenum ores can be very slow, masking the results in short-term laboratory tests. Given that previous ARD/ML studies on-site and elsewhere provided predictions that do not align with long-term site conditions, ARD/ML studies will be ongoing at the Endako Mine through restart and operations.

Characterisation to understand the ARD and ML potential, ARD/ML, and the neutralisation potential of the mined material and waste rock is also required to inform mitigations and meet permitting requirements. ARD/ML predictions form the basis of modelling used for water management and closure planning. Given the outcomes of the previous work, particular attention will be paid to designing the studies to investigate the attenuation and neutralisation potential of all lithologies that will be disturbed by Endako Mine activities.

Characterisation to understand the ARD and ML potential, ARD/ML, and the neutralization potential of the mined material and waste rock is required to inform mitigations and meet permitting requirements. ARD/ML predictions form the basis of modelling used for numerous aspects of water management and closure planning.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL EXPENDITURES

The capital expenditures estimates are based on budget pricing from suppliers for critical components, consultants, contractors and a review of other Canadian projects. Smaller equipment and facilities component costs were factored based on industry norms for the type of facility being constructed and, where possible, adjusted to reflect local conditions. Capital expenditures estimates are within ±40%.

Labour rates are based on contractor costs in the region and country, for similar types of work. Where costs were either not available or irrelevant, costs from other similar projects in Canada were used. The rates used include all cost and profit components payable to contractors.

21.1.1 Basis of Estimates

All expenditure estimates are in 2024 constant Canadian Dollars. The capital expenditures estimates include the following:

- Mine pre-stripping, mining equipment, facilities and fixed and associated consumables and maintenance parts;
- Processing plants new equipment and refurbishing and upgrading of existing equipment and facilities;
- Tailings preparation facility and management facility upgrades and expansion requirements;
- Refurbishing and upgrading infrastructure equipment and materials;
- Construction materials;
- Labour;
- Temporary buildings and services;
- Construction support services;
- Spare parts;
- Initial fills (inventory);
- Freight;
- Vendor supervision;
- Owner’s cost;
- Engineering, Procurement, and Construction Management;
- Commissioning and start up; and
- Contingency.

The capital estimates are based on the following sources with varying degrees of detail and estimates accuracies (PEA to Feasibility study estimates) derived from assessments made from the following sources:

- **Hatch:** Refurbishing and upgrade requirements for processing plant, tailings management facilities and support facilities/infrastructure, and water management and balance;

- **WSP Golder:** Open pit mining related existing facilities and mine rock mechanics;

- **Ausenco:** Filtered tailings plant design and capital costs.

- **AMPL:** All other aspects of capital expenditures estimates based on the AMPL mine and operational plans to an equivalent of PEA accuracy with indicative budget quotes for new and rebuilt equipment.

All expenditure estimates are in the beginning of 2025 constant Canadian Dollars. Exchange rates used are: US$0.74 = CA$1.00.

21.2 DIRECT COSTS

Direct costs are all costs associated with permanent facilities. This includes open pit development, waste rock pre-stripping, equipment and material costs, as well as construction and installation costs.

Mine infrastructure costs for facilities, such as maintenance shops, mine dewatering, etc., were developed based on the plans and general arrangements presented earlier. Expenditures are based on equipment and material actual or quoted costs and contractor quoted installation costs.

Major equipment expenditure estimates are based on the different indicative budget and quotes (as stated in the previous section) obtained from suppliers and installation costs estimated as part of this study.

Direct costs are based on actual takeoffs to the degree developed for this level of study including:

- Earthwork/site work;
- Concrete;
- Structural steel;
- Buildings and architectural;
- Electrical;
- Instrumentation and controls; and
- Piping.

Commodity pricing for earthwork, concrete, steel, architectural and piping is based on local and national costs. Labour rates and equipment usage rates used throughout the estimate were provided by the same source as the commodity prices.

Labour rates generally reflect Canadian present remuneration for the types of work performed.

All labour costs include government mandated contributions and the costs for company provided benefits.

21.2.1 Indirect Costs Estimate

The indirect costs cover all costs associated with temporary construction facilities and services, construction support, freight, vendor representatives, spare parts, initial fills and inventory, owner's costs, EPCM, commissioning and start-up assistance.

The costs for the construction facilities include all temporary facilities, services and operation, site office operations, security buildings and services, construction warehousing and material management, construction power and utilities, site transportation, medical facilities and services, garbage collection and disposal and surveying.

- **Spare Parts:** The cost for spare parts is based on supplier recommendations and budget estimates.

- **Initial Fills (Inventory):** The estimated cost for initial fills is based on 3-months of operating requirements.

- **Freight:** The freight costs were either provided by the vendor or estimated based on weights and typically include containerised and break-bulk shipping and each are respectively divided into ocean freight and inland freight. For imported equipment, the cost of freight and export packing, ex-works to the nearest port, is included with the cost of the equipment. Freight insurance is included in the Owner's cost.

- **Vendor Representatives:** The requirement for the vendor representatives to supervise the installation of equipment or to conduct a checkout of the equipment prior to start-up of the equipment, as deemed necessary for equipment guarantees or warranties, has been included in the estimate. Typically, the cost for this item is inclusive of salary and travel.

- **Taxes and Duties:** Taxes and duties have been included where applicable.

- **Engineering, Procurement and Construction Management (EPCM):** EPCM has been calculated based on the Moon River project managing development and construction and using consultants where deemed appropriate.

- **Capital Cost Qualifications and Exclusions:** All surface construction work will be executed by contractors.

Capital expenditures estimates exclude:

- Sunk costs;
- Taxes and duties;
- Deferred capital;
- Financing and interest during construction;
- Additional exploration drilling;
- Escalation;
- Corporate withholding taxes;
- Legal costs;
- Metallurgical testing costs; and
- Condemnation testing.

21.2.2 Mining

Mine capital cost estimates are based on detailed assessments by Hatch in 2023, for Centerra Gold and as provided to Moon River Moly. The estimates were based on detailed site assessments and quoted pricing from suppliers, consultants and contractors provided with specifications to ensure equipment or service provided is specific to the Project and includes all costs specific to the Project and application. Some small equipment and facilities component costs were factored based on norms for the type of facility being constructed and adjusted to reflect local conditions. The Hatch estimated was adjusted, as required, to be consistent with the present operational plan and inflated by 15% to 2025 constant dollars.

Construction and installation labour rates are based on Owner/Operator costs for the types of work envisaged for the Project.

The mine facilities pre-production capital expenditures are estimated to total $42.1 million including a 20% contingency. The breakdown of the pre-production mine capital expenditures is presented in Table 21.1, below.

TABLE 21.1
MINE PRE-PRODUCTION CAPITAL EXPENDITURES

Description	Mine CAPEX ($)
Pit Electrification	$2,068,000
Sand Plant for Road Building Material	$1,793,000
Second Gyro and Overland Conveyor	$29,769,390
Sub-total Directs	**$33,630,390**
Indirects	**$1,478,000**
Sub-total Direct + Indirects	**$35,108,390**
Contingency (20%)	$7,022,000
Total Project	**$42,130,390**

Additionally, mining equipment will comprise a combination of leased and purchased equipment. The large equipment fleet will be leased by Moon River while small equipment and vehicles will be purchased. The total equipment leasing down payment and equipment purchases total $23.3 million including a contingency of 20%.

21.2.3 Processing Plant

Processing plant rehabilitation and upgrading capital cost estimates are based on detailed assessments by Hatch in 2023, for Centerra and as provided to Moon River, as well as AMPL estimates for reconfiguring and refurbishing the Old Plant grinding circuit. The estimates were based on detailed site assessments and quoted pricing from suppliers, consultants and contractors provided with specifications to ensure equipment or service provided is specific to the Project and includes all costs specific to the Project and application. Some small equipment and facilities component costs were factored based on norms for the type of facility being constructed and adjusted to reflect local conditions. The Hatch estimates were adjusted, as required, to be consistent with the present operational plan and inflated by 15% from 2023 to 2025 constant dollars.

Construction and installation labour rates are based on contractor costs for the types of work to be performed.

Total pre-production capital expenditures for the processing plant is $106.4 million including a 20% contingency. The breakdown of the pre-production capital expenditures is presented in Table 21.2, below.

TABLE 21.2 PROCESSING PLANT PRE-PRODUCTION CAPITAL EXPENDITURES	
Description	**Total Cost – 2025 ($)**
Process Building	$45,018,000
Process Building – Extension	$6,399,000
Mill Expansion, CAPEX, Non Rod Mill Reconfiguration	$9,383,516
Rod Mill Configuration	$3,228,504
Sub-total Directs	**$64,029,020**
Construction Support	$13,711,000
Indirects	$11,426,000
Sub-total Directs and Indirects	**$89,166,020**
Contingency	$17,253,000
Processing Plant CAPEX Total	**$106,419,020**

21.2.4 Tailings Management Facility

Tailings management facility preproduction capital estimates were developed by Ausenco Engineering Canada Inc.

The tailings management facility pre-production expenditures are estimated to be $150.1 million.

21.2.5 Water Management and Water Treatment

The water management and water treatment preproduction capital estimates were developed by Ausenco Engineering Canada Inc.

Pre-production capital costs for overland water conveyance, including diversion channels, ponds, pumping and piping is $31.5 million, based on first principles estimates of required conveyance volumes, flow rates and pumping head. The capital expenditures estimate for effluent water treatment plant was benchmarked against capital costs for existing plants treating similar influents and contractor quotes received for similarly sized projects (*i.e.*, influent flow rates of approximately 1,250 m^3/hr). The estimated capital cost for water treatment is $52.7 million.

21.2.6 Project Indirects and Owner's Costs

Project Indirects and Owner's costs are estimated at $10.0 million over the pre-production period. Owner's costs also include all equivalent G&A costs, which would be incurred during the construction phase.

21.2.7 Total Capital Expenditures

Pre-production capital expenditures for the restart project are estimated to total $493.7 million. The total capital expenditures include a 20% contingency. The breakdown of the capital expenditures is presented in Table 21.3, below.

TABLE 21.3 PRE-PRODUCTION CAPITAL EXPENDITURES ESTIMATES	
Component	**Total Expenditures ($ million)**
Mine	$35.1
Equipment Lease Deposit and Purchases	$23.3
Processing Plant	$89.2
Tailings Management Facilities	$150.1
Surface Infrastructure	$13.2
Non- Mining Mobile Equipment	$5.0
Water Management	$31.5
Water Treatment	$52.7
Owner's Costs	$10.0
Contingency	$83.7
Total	**$493.7**

In addition to the capital expenditures, working capital of $57.2 million has been estimated based on 3 months of operating costs.

21.3 SUSTAINING CAPITAL

Sustaining capital requirements are estimated of $3.2 million comprising expanding the filtration circuit for the dry stack tailings production, rebuilds and new purchases of mining equipment, and upgrades for the processing plant and surface infrastructure. The sustaining capital expenditures summary for the two mines is shown in Table 21.4, below.

TABLE 21.4
SUSTAINING CAPITAL EXPENDITURES SUMMARY FOR THE TWO MINES

	Year										Total
	1	2	3	4	5	6	7	8	9	10	
Mine											**$0**
Equipment Lease Deposit and Purchases											**$0**
Processing Plant		$240,000	$240,000	$240,000	$240,000	$240,000	$240,000				**$1,440,000**
Tailings Management Facilities											**$0**
Surface Infrastructure		$120,000	$120,000	$120,000	$120,000	$120,000	$120,000				**$720,000**
Non-Mining Mobile Equipment											**$0**
Water Management											**$0**
Water Treatment		$60,000	$60,000	$60,000	$60,000	$60,000	$60,000				**$360,000**
Owner's Costs											**$0**
Contingency (25%)		$105,000	$105,000	$105,000	$105,000	$105,000	$105,000				**$630,000**
Total Capital Expenditures	**$0**	**$525,000**	**$525,000**	**$525,000**	**$525,000**	**$525,000**	**$525,000**	**$0**	**$0**	**$0**	**$3,150,000**

21.4 OPERATING COST ESTIMATES

21.4.1 Basis for Estimates

Project operating costs, by department, are based on estimates provided by Hatch and WSP Golder (adjusted from 2022 to 2025 prices) for the *uncompleted* Feasibility Study by Centerra. Operating costs for infrastructure support facilities and services were adjusted to reflect changes included in the proposed operating plan of AMPL.

Project departmental operating costs were divided into two components – consumables/maintenance parts and labour. The consumables component includes all materials and parts needed for mining, processing and surface facilities and the operation and maintenance of equipment for these areas. Actual costs for consumables were obtained from suppliers. Maintenance parts and consumables are based on equipment suppliers. The total mine labour force complement and salaries were calculated on a total yearly basis. The labour component was combined with the materials component to produce the yearly departmental operating cost estimates.

The G&A cost components include the materials and supplies used by the administration and surface services groups. These costs comprise office supplies, computer supplies and computer and software upgrades, light vehicle and surface equipment operating and maintenance consumables, business travel, fees for consultants and communications costs.

Labour costs and salaries for all services for labour and mine staff have been estimated on a yearly total cost basis.

Critical operating cost components are based on the following costs:

- Diesel fuel price of $1.75/litre; and
- Electrical power cost of $0.065/kW.

Labour rates are based on local rates from other mining operations and contractor costs. The contractor rates used include all cost and profit components payable to the contractors.

All costs are quoted in constant 2025 Canadian Dollars.

21.4.2 Mining Cost

Mine operating cost estimates were developed from first principles using site historical costs, costs from similar mines in the region, costs from similar mines operated by Centerra Gold and costs from Golder's proprietary database, as a basis for calibrating the operating cost models. This included, among others, detailed estimates of personnel for all required functions, equipment maintenance and consumption, ancillary equipment needs and expected operating hours.

Mine OPEX were based on key inputs that include but are not limited to assumed material movement; equipment productivities; fuel, lube, tires and parts consumption; maintenance, labour, blasting consumables and parts costs; and ancillary equipment required to support the operations of the primary equipment. Costs were estimated for each mining unit operation including drilling, blasting, loading and hauling, maintenance, support and general mine and engineering. The cost model uses the same mining equipment as set out for the LOM Plan, specified in Section 16.8. The mine operating costs are summarised, by activity, in Table 21.5, below.

TABLE 21.5
MINE OPERATING COST ESTIMATE SUMMARY BY ACTIVITY

Activity	PEM[1] Cost ($/t)	Waste Mining Cost ($/t)
General Mine and Engineering	0.17	0.17
Drilling	0.07	0.05
Loading	0.40	0.37
Haulage	1.05	1.01
Maintenance	0.30	0.29
Blasting	0.35	0.30
Support	0.20	0.20
Total Endako Mine Operating Cost	**2.54**	**2.39**

[1]Potentially Economic Mineralisation.

21.4.3 Processing Cost

The operating cost for the process plant was developed based on the planned throughput of 7,000 tonnes per day. The operating cost estimate comprises of process plant consumables, power, maintenance and labour.

The average annual process plant operating cost is $ per tonne processed. Table 21.6, below, provides the summary processing cost breakdown by major area.

TABLE 21.6
PROCESSING AND TAILINGS MANAGEMENT COSTS

Component	Cost
Manpower	$0.64
Mill Reagents/Consumables	$1.93
Environmental	$0.48
Power	$1.40
Total Annual Mill OPEX	**$4.45**

21.4.4 Summary of Operating Cost Estimate

The operating cost estimate for effluent treatment at the water treatment plant was benchmarked against operating costs for existing plants treating similar influents and contractor quotes received for similarly sized projects (*i.e.*, influent flowrates of approximately 1,250 m^3/h).

The estimated annual cost for water treatment is C$2.1 million during initial pit dewatering in Year -1, equivalent to C$0.01/tonne milled over the LOM. During initial dewatering (*i.e.*, Year -1) the annual operating cost of overland water conveyance is $1.9 million.

During operating Years 1 to 10, the annual operating cost for overland water conveyance is $1.5 million, including pump operating costs and maintenance.

21.4.5 Surface Department

The surface department operating costs total $9.1 million with the breakdown shown in Table 21.7, below.

TABLE 21.7
SURFACE DEPARTMENT OPERATING COSTS

Description	Total Cost ($)
Surface Hourly	$5,589,000
Surface Staff	$1,363,500
Surface Equipment	$2,128,000
Total Surface Department	**$9,080,500**

21.4.6 General and Administration (G&A) Costs

The estimates for G&A costs encompass all operating costs associated with operating the offices and providing materials and supplies for staff functions.

The total yearly G&A costs are estimated to be approximately $10.8 million (presented in Table 21.8, below), of which approximately $4.5 million is for salaries and benefits. Employee burdens account for approximately 35% of the total salary for each employee. Annualised site G&A, Surface Department and Water Management costs are estimated at $0.86 per tonne of potentially economic mineralisation processed.

The mine management and administration roster and costs have been estimated in Table 21.8, below. A total of 42 people would be employed in this area, most of which would be staff positions. They would be responsible for the management, administration, personnel, accounting, purchasing needs and distribution of material to the operation, site security, health and safety and environmental issues. The total costs for G&A labour are $0.18 per tonne of potentially economic mineralisation processed.

TABLE 21.8 GENERAL AND ADMINISTRATIVE (G&A) COSTS	
	Per Year
Site Salaries and Overhead	$4,548,000
Communications/Internet	$120,000
Equipment Rental and Maintenance	$50,000
Administration – Non-labour	$69,000
Accounting – Non-labour	$173,000
Warehouse Supplies	$30,000
Postage, Courier and Light Freight	$100,000
Insurance	$1,800,000
Property and Municipal Taxes	$1,200,000
Permits and Licences	$125,000
Bonding and Bank Charges	$500,000
Technical Consultants	$100,000
Professional Fees – Accounting	$180,000
Professional Fees – Legal	$200,000
Human Resources – Non-labour	$154,000
Recruitment	$40,000
Security Supplies	$50,000
Safety, Clothing and Training	$100,000
First Aid	$75,000
Dues and Subscriptions	$15,000
Public Relations	$100,000
Environmental	$0
Power	$215,000
Surface Transportation – Pickups (Leases)	$60,000
Professional Fees – General	$15,000
Travel and Accommodation – Business	$30,000
Freight	$700,000
Miscellaneous	$50,000
Total G&A Costs	**$10,799,000**

21.4.7 Concentrate Transport Charges

Transportation charges of $374 per tonne of concentrate have been included in the cash flow model.

21.5 PROJECT TOTAL OPERATING COSTS

The estimated Base Case total average operating cost (excluding smelting and refining) is approximately $11.84 per tonne of ore and the equivalent of US$11.61/lb of molybdenum. Table 21.9, below, presents a summary table of the LOM average operating costs for each department on a cost per tonne of ore basis.

TABLE 21.9
PROJECT OPERATING COSTS SUMMARY – BASE CASE

Component	Cost/tonne ($)
Mining	$5.45
Processing and Tailings	$5.53
Surface Department, Environmental and G&A	$0.86
Total Operating Cost per Tonne Ore	**$11.84**
Total Operating Cost per Pound of Molybdenum	**US$11.61**

21.6 EXCLUSIONS

For the purpose of this study, value added taxes (as operating mines can claim and be reimbursed VAT for capital and operating expenses in the year incurred) and other taxes, along with import duty costs, have not been included. Exploration costs and all costs associated with areas beyond the Property limits have also not been included.

22.0 FINANCIAL ASSESSMENT

The expected cash flow estimates are calculated using the forecast mine plan, operating costs and capital expenditures incorporating the 2-year moving average molybdenum oxide price.

The forecast financial returns included a molybdenum oxide price of US$49.73/kg ($22.50/lb).

22.1 KEY ASSESSMENT PARAMETERS

The financial evaluation was carried out for the Base Case using the key parameters presented in Table 22.1, below.

TABLE 22.1
BASE CASE PARAMETERS

Parameter	
Molybdenum Oxide Price	US$49.73/kg ($US22.50/lb.)
Potentially Economic Mineralisation Mined	273 million tonnes grading 0.075% MoS_2
Waste Mined	186 million tonnes
Unplanned Mining Dilution	5%
Projected Mining Recovery	90%
Recovery	75.7%
Average Processing Rate	27 million tonnes per annum
Payable Molybdenum Produced Annually	9.2 million kg
Initial Acquisition Cost and CAPEX Funding	US$493.7 million
Working Capital	US$57.2 million
Total Sustaining Capital Expenditures	US$3.2 million
Closure Cost	US$40.0 million
Estimated Operating Costs ($ per tonne)	
Mining (per tonne potentially economic mineralisation)	US$5.45
Processing and TMF	US$5.53
G&A	US$0.86
Life-of-Mine	10.2 years

Revenue is based on payables terms and smelter and refining costs for a third-party smelter.

Costs for metal sales and shipping are included in the deductions that the refiner makes.

Capital expenditures, as shown in the capital section, would be incurred over an approximate 1.5 to 2 year period, which is reflected in the discounted cash flow calculations. The cash flows include sustaining capital.

Operating costs, as presented in the operating costs section, are incorporated. Mining equipment leasing is included in the operating costs.

All aspects of the operations are carried out by company personnel.

Working capital, first fills, and spare parts have been included in the upfront capital amount, with ongoing working capital movements estimated based on maintaining a rolling quarterly balance equivalent to one month of total operating expenses. Working capital is returned to a nil balance at the end of the mine life.

Provision has been made for depreciation using the straight-line method over the LOM for initial and sustaining capex.

The British Columbia and Canada Corporate Income Tax rates are 11% and 15%, respectively.

The British Columbia Mining Tax rate is 2% on taxable profit until pre-production capital expenditures are recovered when it increases to 13% of taxable profits.

22.2 FINANCIAL RETURNS

The overall level of accuracy of this study is ±20% for the processing plant and surface infrastructure components and ±25% to ±30% for all other components.

The financial returns from the potentially economic mineralisation are presented for the expected parameters and costs at a molybdenum price of $US49.73/kg ($US22.50/lb.) of molybdenum oxide is presented in Table 22.2, below.

TABLE 22.2
FINANCIAL ASSESSMENT EXPECTED RETURNS

	Pre-Tax	After-Tax
Pre-production CAPEX ($ millions)	$493.7	$493.7
Undiscounted Net Revenue ($ millions)	$5,854	$5,854
Undiscounted Total Cash Flow ($ millions)	$2,087	$1,478
NPV (5%) – millions	$1,405	$996
NPV (8%) – millions	$1,116	$790
IRR	46%	40%
Payback Period	2.2 years	2.2 years

Results show a reasonable return on the pre-production capital investment of $493.7 million.

The expected average operating cost over the Endako Mine LOM is $11.84 per tonne of potentially economic mineralisation or US$11.61 per pound of molybdenum oxide.

22.3 SENSITIVITY ANALYSIS

Several sensitivities were investigated to determine the effect, on the key financial statistics, of 5%, 10%, 15% and 20% positive and negative variances in the following parameters:

- Mined Grade;
- Molybdenum Price;
- Operating Cost;
- Capital Expenditures; and
- US$:CA$ Exchange Rate.

The results of the sensitivity analysis are presented in Table 22.3 and Table 22.4, below. The returns from the operations are most sensitive to metal price, exchange rate and mined grade variations and least sensitive to capital expenditures and operating costs changes.

TABLE 22.3
SENSITIVITY ANALYSIS FOR AFTER-TAX NPV

Parameter	After-Tax NPV 8% ($ million)								
	-20%	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**
Mined Grade	114	313	473	635	790	945	1,100	1,254	1,409
Molybdenum Price	-302	-31	265	526	790	1,061	1,344	1,641	1,948
Operating Costs	1,071	1,001	931	862	790	719	649	573	502
Capital Costs	865	846	828	808	790	771	753	734	716
US$:CA$ Exchange Rate	273	407	536	666	790	915	1,038	1,161	1,284

TABLE 22.4
SENSITIVITY ANALYSIS FOR AFTER-TAX IRR

Parameter	After-Tax IRR (%)								
	-20%	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**
Mined Grade	14	21	28	34	40	46	52	58	64
Molybdeum Price	-7	7	19	30	40	51	62	73	84
Operating Costs	52	49	46	43	40	37	35	31	28
Capital Costs	50	48	45	43	40	38	37	35	33
US$:CA$ Exchange Rate	19	25	30	36	40	45	50	55	59

Figure 22.1 and Figure 22.2, below show the sensitivity analysis for after-tax NPV8% and IRR in graphical form.

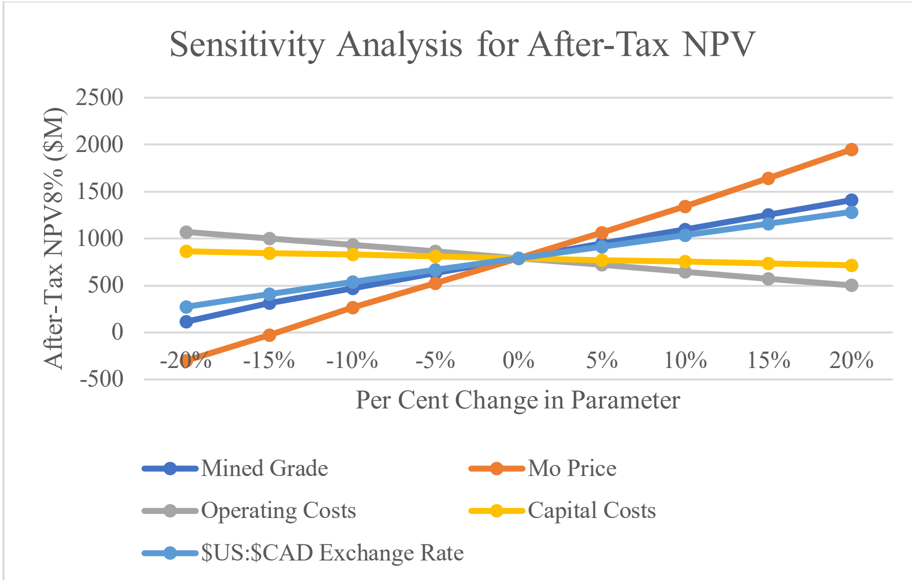


Figure 22.1 Graph of NPV Sensitivity Analysis at 8% Discount Rate
Source: *AMPL, 2025*

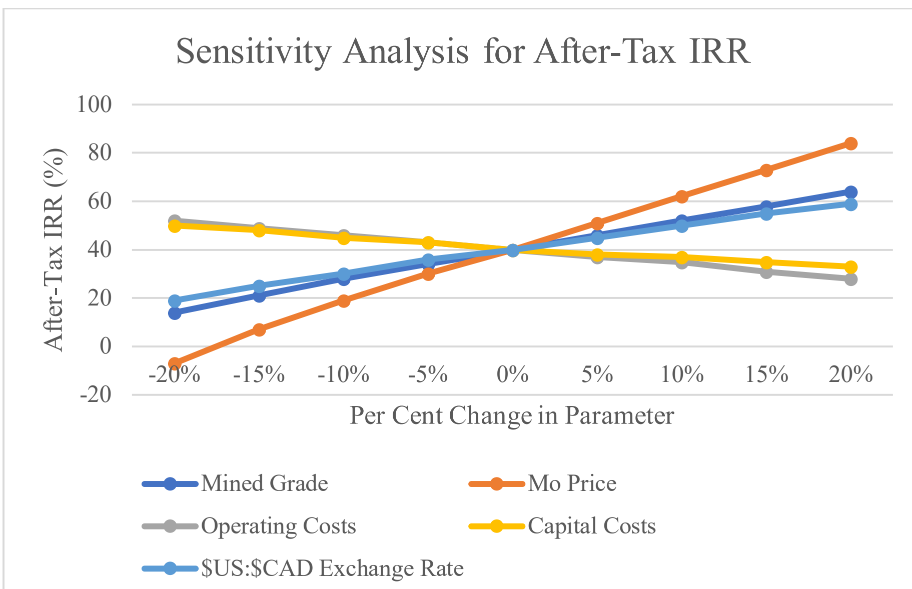


Figure 22.2 Graph of IRR Sensitivity Analysis
Source: *AMPL, 2025*

23.0 ADJACENT PROPERTIES

There are no relevant adjacent properties.

24.0 OTHER DATA AND INFORMATION

There is no other data or relevant information.

25.0 INTERPRETATION AND CONCLUSIONS

This Preliminary Economic Assessment (PEA) examines the viability of restarting open pit mining and flotation processing of the Mineral Resources reported in this PEA Technical Report. The results from this PEA indicate the Endako Project can generate positive economic returns.

The resources of the Endako Mine comprise of Measured, Indicated and Inferred Mineral Resources. It should be noted that the Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. Therefore, there is no guarantee that the economic projections contained in this PEA would be realised.

Using a cut-off grade of 0.04% MoS_2, there is a Measured and Indicated Resource of 336 million tonnes at 0.072% MoS_2 and an Inferred Resource of an additional 60 million tonnes at 0.054% MoS_2 available for mining. This PEA has identified a diluted potentially mineable resource of 273 million tonnes at an average grade of 0.075% MoS_2 (0.045% Mo) after dilution and mining losses.

The restart plan includes mining and processing 75,000 tonnes per day of potentially economic mineralisation with refurbishing and reconfiguration of the 2 existing processing plants on site (the New Plant on care and maintenance and the Old Plant mothballed).

The existing tailings management facility consisting of TP-1 and TP-3 will be re-commissioned and merged for tailings storage using the existing method of water cover of disposed tailings. The existing tailings dams will be increased in height, using hydrocycloned tailings course fractions, to accommodate the planned potentially economic mineralization mining and processing rate.

The support infrastructure and facilities are all existing and requiring some upgrading and refurbishing.

The overall expected economic results indicate the Project is economic, generating undiscounted cashflows (over the 10-year LOM) of approximately \$2.1 billion and \$1.5 billion on a before and after-tax basis, respectively.

At the forecast molybdenum metal price (US\$49.73/kg or US\$22.50/lb of Mo), the Project's after-tax $NPV_{8\%}$ is estimated to be approximately \$790 million. The after-tax Internal Rate of Return (IRR) is estimated to be 40%, with payback of approximately 2, 2 years from start of production.

Sensitivity analysis shows that the Project economics are most sensitive to mined grade, exchange rate and metal price. The Project is least sensitive to capital expenditures.

Based on the study results, AMPL concludes that the Endako Mine restart provides positive returns based on the parameters and metal prices used in this study.

The Endako Mine restart plan should immediately progress to a Feasibility Study to support financing of the restart of operations.

26.0 RECOMMENDATIONS

Based on the AMPL assessment for the Endako Mine restart, recommendations follow:

1. Complete a Feasibility Study for mine restart using a mining and processing rate of approximately 75,000 tonnes per day (27 million tonnes per annum) of potentially economic mineralisation. A new block model using metric units will be required. (Estimated cost is $4-6 million.)

2. Develop an updated or new detailed water management model and Tailings Management Facility (TMF) design, which would include hydrology and TMF leachate seepage data and forecasts. (Estimated cost $500,000.)

3. Initiate renewal of permits to encompass the proposed operations immediately upon the initiation of further studies.

4. Engage with area Indigenous rightsholders and the Water Quality Working Group on the proposed plans early in the Feasibility Study process.

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Geotechnical Pit Wall Stability Assessment of the Denak Pit, November 3, 2009, Golder Associates

Technical Report Endako Molybdenum Mine, John M. Marek, P.E., Independent Mining Consultants, Inc., September 12, 2011

Technical Report on the Endako Mine (Incomplete), British Columbia, Canada, WGM/Centerra Gold Inc., 2018

Endako Restart Project Engineering Report, Hatch, 2022

Basis of Estimate – Endako Restart Project, Hatch, 2022

Feasibility Mine Planning Study for Endako Mine, WSP-Golder, May 2022

Water Balance Model for the Endako Mine Restart Report – DRAFT, SRK Consulting (Canada) Ltd., August 2022

Water Treatment Design Basis for the Endako Mine Restart Memorandum, SRK Consulting (Canada) Ltd., June 2022

Pump and Pipeline Design Basis for the Endako Mine Restart Water Management Memorandum, SRK Consulting (Canada) Inc., June 21, 2022

Feasibility Design of Tailings Ponds 1 and 3 for Life of Mine Plan, Endako Mine, WSP-Golder, May 26, 2022

CERTIFICATE OF QUALIFICATIONS

I, Brian C. LeBlanc, P.Eng., residing in Thunder Bay, Ontario, P7G 1M6, Canada, do hereby certify that:

1. I am President and a Principal at A-Z Mining Professionals Ltd.
2. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Endako Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of November 21, 2025.
3. I am a graduate of Michigan Technological University in 1986 with a Degree in Mining Engineering.
4. I am licensed by the Professional Engineers of Ontario, Registration No. 904279972-10.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for the purposes of NI 43-101.
6. My relevant experience for the purpose of the Technical Report is:
   a) Extensive and progressively more senior engineering and operational duties at base metals, gold and nickel mining operations and development projects.
   b) 16 years of experience directing and overseeing several scoping level, pre-feasibility level, and feasibility level studies for mines and mining companies.
   c) Mill Operator – Giant Yellowknife Mines, 1974-1975
   d) Crusher Operator/Screening Plant Operator/Loadout Operator/Surveyor – Steep Rock Iron Mines Ltd., 1976-1979
   e) Mine Planner/Chief Surveyor – Nanisivik Mines Ltd., 1981-1984
   f) Mining Engineer/Underground Supervisor/Mine General Foreman/Technical Services Superintendent/Mine Superintendent – Williams Mine, 1986-2003
   g) Manager of Mining – Kinross' Kubaka Mine (Russia), 2003-2004
   h) Technical Services Superintendent – Lac Des Isles Mines, 2004-2006
   i) Project Superintendent – Redpath Indonesia, 2006-2007
   j) Project Manager for Ontario – North American Palladium Ltd., 2007-2010
   k) General Manager/Vice President/President – NordPro Mine & Project Management Services Ltd, 2010-2014
   l) President, A-Z Mining Professionals Limited, February 2014 to Present
7. I assisted in preparation of the Technical Report and Peer Review for Sections 1.0 thru 26.0. I co-authored Sections 1.0, 2.0, 3.0, 25.0 and 26.0 of the Technical Report and take responsibility for the report.
8. I have completed a personal inspection of the Property that is the subject of the Technical Report.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of both the issuer and the vendor applying all the tests in Section 1.5 of NI 43-101.
11. I have not had prior involvement with the Property that is the subject of the Technical Report.
12. I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Effective Date: November 21, 2025

Signing Date: January 5, 2026

Original Signed and Sealed by
Brian LeBlanc, P.Eng.
Registration No. 904279972-10

CERTIFICATE OF QUALIFICATIONS

I, Finley Bakker, P.Geo., residing in Campbell River, B.C., V9H-1C6, Canada, do hereby certify that:

1. I am a Professional Geoscientist with EGBC (1991) in the Province of British Columbia Registration No. 18,639 at Finley Bakker Consulting, Permit Number 1003901.
2. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Endako Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of November 21, 2025.
3. I am a graduate of McMaster University with a Hons. Bachelor of Science in Geology (1979)
4. I co-authored and assisted in the preparation of Sections 1.0, 2.0 and 3.0 and am responsible for Sections 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0 and 14.0.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI-43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for the purposes of NI 43-101.
6. My relevant experience for the purpose of the Technical Report is:
    a) Practiced my profession continuously since 1979.
    b) Thirty-seven (37) years of experience utilising MineSight™/HxGN Mine Plan™ 3D software.
    c) Forty-four (45) years of experience calculating Resources and Reserves including calculating Resources for Davidson Molybdenum Property and Stuhini Exploration (Molybdenum Project).
    d) Chief Geologist at 4 mines including Chief Geologist at Adanac Molybdenum Corp. Ltd.
    e) Have also held the positions of Senior Resource Geologist, Exploration Manager and Superintendent of Technical Services.
7. I have completed a personal inspection of the Property that is the subject of the Technical Report on April 9, 2025 and September 9-12, 2025.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
10. I am also independent of the Vendor and the Property.
11. I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Effective Date: November 21, 2025

Signing Date: January 5, 2026

Original Signed and Sealed by
Finley Bakker, P.Geo.
Registration No. 18,639
Permit Number 1003901

PROFESSIONAL
PROVINCE OF
F. H. BAKKER
BRITISH COLUMBIA
GEOSCIENTIST

CERTIFICATE OF QUALIFICATIONS

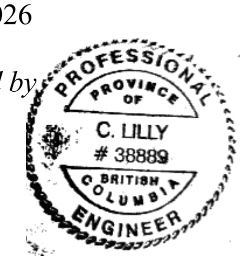
I, Cameron W. Lilly, 3054 Dog Creek Road, Williams Lake, BC, V2G 4X2, do hereby certify that:

1. I am a registered, professional engineer of the Engineers and Geoscientists British Columbia, license number 38889.
2. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Endako Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of November 21, 2025.
3. I graduated from the University of British Columbia in 2001 with a Bachelor's of Applied Science specializing in Mineral Process Engineering.
4. I am responsible for Sections 13.0 and 17.0.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for the purposes of NI 43-101.
6. My relevant experience for the purpose of the Technical Report is:
   a) I have practiced my profession continuously since 2001.
   b) I have worked in mining for 24 years and have been P.Eng for the last 12 years.
   c) I have held positions, such as Plant Senior Metallurgist, Plant Shift Supervisor, Mill Manager and Project Engineer as well as Process Engineer with consulting engineering firms.
7. I completed a personal inspection of the Property that is the subject of the Technical Report on September 10-11, 2025.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
10. I am independent of the Vendor and the Property.
11. I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Effective Date: November 21, 2025

Signing Date: January 5, 2026

Original Signed and Sealed by
Cameron W. Lilly, P.Eng.
Registration No. 38889



CERTIFICATE OF QUALIFICATIONS

I, Will Coverdale, at Unit D, 9F, Kai Centre, 36 Hung To Road, Kwun Tong, Kowloon, Hong Kong, do hereby certify that:

1. I am a Mining Engineer (independent consultant/at Coverdaleco).
2. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Endako Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of November 21, 2025.
3. I am a graduate of the University of Queensland, Australia in 2004 with a Bachelor of Mining Engineering.
4. I am a member of the Australasian Institute of Mining and Metallurgy (AusIMM) (Membership No. 211261).
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for the purposes of NI 43-101.
6. My relevant experience for the purpose of the Technical Report is:
   a) Over 20 years of global experience in the minerals industry, including operational, technical, and corporate roles across multiple commodities (including experience through broad base and industrial minerals exposure) and mining methods.
   b) Extensive experience in open-pit mine planning, feasibility studies, reserve estimation, technical due diligence, and mine optimisation for open-pit operations (*e.g.*, Project Engineer roles on large-scale open-pit projects including Masbate (Philippines) and Olaan Ovu (Mongolia); delivery of scoping and feasibility studies incorporating open-pit options for projects such as Chaah (Malaysia), Koka (Eritrea), Romang Island (Indonesia), Westmorland (Northern Territory, Australia), and Wallbrook/Pinnacles (Western Australia, Australia)).
   c) Preparation of numerous NI 43-101 and JORC-compliant technical reports, reserve estimates, and independent qualified person reports, including technical due diligence and valuations for projects globally.
   d) International experience, with direct consulting on Canadian mineral projects, combined with broad commodity exposure including industrial minerals and base metals projects.
7. I authored and assisted in preparation of the Technical Report and take responsibility for Section 16.0. I also co-authored Sections 1.0, 19.0, 21.0, 22.0, 25.0 and 26.0.
8. I have not completed a personal inspection of the Property that is the subject of the Technical Report.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
11. I have not had prior involvement with the Property that is the subject of the Technical Report.
12. I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Effective Date: November 21, 2025

Signing Date: January 5, 2026

Original Signed and Sealed by
Will Coverdale, P.Eng.

CERTIFICATE OF QUALIFICATIONS

I, Scott C. Elfen, P.E., P.Eng., certify that I am employed as a Global Lead Geotechnical and Civil Services with Ausenco Engineering Canada ULC (Ausenco), with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC, V6E 3S7, Canada.

1. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Endako Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of November 21, 2025.
2. I graduated from University of California, Davis, California, USA in 1991 with a Bachelor of Science degree in Civil Engineering (Geotechnical).
3. I am a Registered Civil Engineer with the State of California (License No. C56527) by exam since 1996. I am also a member in good standing of Engineers and Geoscientists British Columbia (Registration No.64064).
4. I have practiced my profession continuously for 28 years with experience in the development, design, construction, and operations of mine waste storage facilities, such as waste rock storage facilities and tailings storage facilities ranging from slurry to dry stack facilities, focusing on precious and base metals, both domestically and internationally.
5. I have developed geotechnical design parameters for pit slope design, plant foundation design, and other supporting infrastructure. Examples of projects I have worked on include Skeena's Eskay Creek Project PEA, PFS, and FS, O3 Mining's Marban Project PEA and PFS, First Mining Gold's Springpole PEA and PFS. SSR Mining's Puna Silver In-Pit Tailings Disposal PFS, and Detailing Engineering, and Lumina Gold Corp's Cangrejos Project PEA and PFS.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Endako Mine Project on September 15, 2025 for a visit duration of one day.
8. I am responsible for Sections 18.3, and 21.2.4 of the Technical Report.
9. I am independent of Moon River Moly Ltd., as independence is defined in Section 1.5 of NI 43-101.
10. I have had no previous involvement with the Endako Mine Project.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Effective Date: November 21, 2025

Signing Date: January 5, 2026

Original Signed and Sealed by
Scott C. Elfen, P.E., P.Eng.

CERTIFICATE OF QUALIFICATIONS

I, Jonathan Cooper, P.Eng., certify that I am employed as a Water Resources Engineer with Ausenco Sustainability ULC ("Ausenco"), with an office address of 11 King Street West, Suite 1500, Toronto, Ontario, M5H 4C7, Canada.

1. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Endako Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of November 21, 2025.
2. I graduated the University of Western Ontario with a Bachelor of Engineering Science in Civil Engineering in 2008, and University of Edinburgh with a Master of Environmental Management in 2010.
3. I am a Professional Engineer registered and in good standing with Professional Engineers Ontario (Registration No. 100191626), Engineers and Geoscientists British Columbia (Registration No. 37864) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (Registration No. L4227).
4. I have practiced my profession continuously for over 15 years with experience in the development, design, operation, and commissioning of surface water infrastructure. Previous projects that I have worked on that have similar features to the North Island Project are the Kwanika-Stardust for NorthWest Copper, located in British Columbia, Colomac Gold Project located in the Northwest Territories, and the Crawford Project located in Ontario.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Endako Mine Project.
7. I am responsible for Sections 18.4 and 21.2.5 of the Technical Report.
8. I am independent of Moon River Moly Ltd., as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with Endako Mine Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Effective Date: November 21, 2025

Signing Date: January 5, 2026

Original Signed and Sealed by
Jonathan Cooper, P.Eng.