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PREPARED FOR:

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CONCERNING:

ScoZinc Mine
Preliminary Economic Assessment Update

Document No. RPT-17698-0001
Preliminary Economic Assessment
Revision 0

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1.0 SUMMARY

Stantec Consulting Ltd. (Stantec), produced this technical report to update the Preliminary Economic Assessment (PEA) of restarting the ScoZinc mine. The previously-updated PEA, dated 12 June 2013, provided a revised open pit mine plan confirming a significant increase in mine life for the Main and Northeast deposits. This PEA update builds on that mine plan and incorporates a proposed underground mining operation between the Main and Northeast open pits, and blending of the higher-grade material with the lower-grade open pit mineralization in years 5 to 7 of the mine plan. Updated equipment, capital and operating cost estimates from a variety of sources are also included in the PEA, along with a review of historic milling performance and data. The report also addresses the known historic challenges and risks which impacted production during previous operating periods. The open pit will be managed using contract resources for the life of open pit operations.

This technical report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

1.1 Economic Analysis

The potential economic viability of the project was evaluated using a discounted cash flow analysis approach. In summary, based on the base case metal pricing assumption, the results of the preliminary economic analysis indicate that:

- Based on a mill throughput rate of 2,600 tonnes per day (t/d), the project has a mine life of approximately 7.7 years, and offers an approximate payback period of 1.9 years;
- Combined open pit and underground mine operating costs of C$41.31/t milled;
- Mill operating cost of C$13.46/t milled;
- Mine and mill restart capital expenditures (CAPEX) of C$26.9 million (including $1.34 million in contingency);
- Assuming base case zinc (Zn) and lead (Pb) prices of US$1.25 and US$1.05/lb, respectively, and an exchange rate of C$1.00 to US$0.81, the project has an estimated pre-tax internal rate of return (IRR) of 67.3% and an after-tax IRR of 63.7%;
- The project has a pre-tax net present value (NPV) of $159.9 million and an after-tax NPV of $127.9 million, both using a 5% discount rate. At an 8% discount rate, the pre-tax NPV is $134.2 million and the after-tax NPV is $107.7 million.
- Direct C1 zinc cash cost of production (after deducting credits for lead) for the life-of-mine (LOM) is C$0.73/lb (US$0.59/lb).

- Earnings before interest, taxes, depreciation, and amortization (EBITDA) for LOM averages C$31.1 million per annum.
- Total payable metal production over the life of the project is projected to be 323 million lb (147,008 t) of Zn and 184 million lb (83,657 t) of Pb.
- Total LOM gross revenue is estimated to be $645 million, 67% of which is derived from Zn and 33% derived from Pb.

The cash flow model is based on a scenario in which two open pits are to be mined sequentially and blended with feed from an underground mining operation. The two open pits are the Main (including the Southwest Expansion and Tadpole) and Northeast, each of which have been optimized using pit optimization software. The underground mine targets the upper higher-grade portion of the mineral resource between the Main and Northeast pits, lying beneath the highway and Gays River, and will contribute to the mill feed in Years 5, 6, and 7 of the LOM plan.

Note that the Getty pit, included in a previous PEA, is not included in this analysis but provides potential for additional mine life.

The production scheduling is based on mill feed provided from two open pits, an underground mining operation and stockpiles, with an average production rate of 852,800 tonnes per year (or 2,600 t/d) over an average of 328 operating days per year. Aggregate production from the open pits and the underground mine is estimated at 6,552,000 t grading 3.06% Zn and 1.57% PB.

The average strip ratio for the open pits LOM is 12.0-to-1.0 (excluding pre-stripping, which is included in the capital costs). Approximately 62% of the open pit waste is assumed to be readily removed without blasting, including soils that will be used for reclamation. Open pit mine dilution is assumed to be 7.5% at grades of 1% Zn and 0.5% Pb. Mining losses are assumed to be 5%.

The underground operation will be accessed from the lower benches of the open pits to reduce waste development costs and to use the open pit excavations and facilities for water management. The underground workings and related facilities will be designed to produce up to 500 t/d of higher-grade feed to the mill to blend with the lower-grade mill feed from the ongoing open pit operations. Diluted and recoverable underground mineral resources are estimated at 289,300 t grading 6.54% Zn and 3.74% Pb.

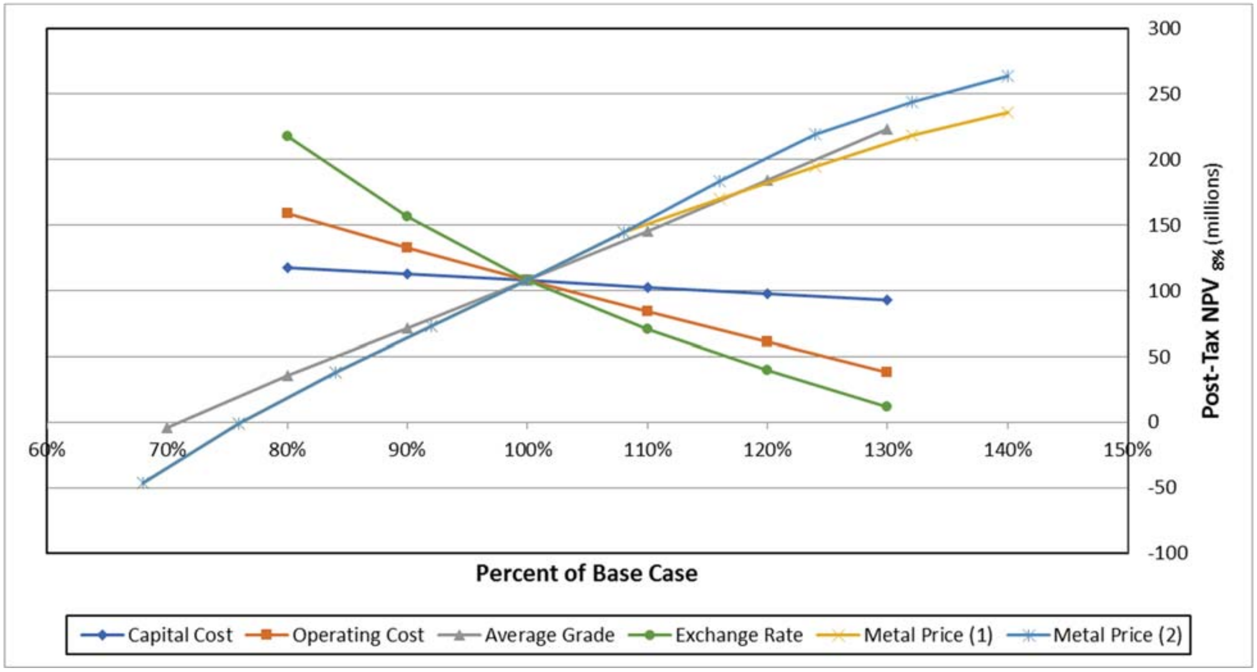
The proposed open pits and underground mine contain the potentially mineable resources, termed mill feed, with classifications having the meanings ascribed to them by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council. The proposed production schedule is based on milling a total tonnage of 6.55 million tonnes over the life of the project, of which, about 9.8% is in the Inferred category.

ScoZinc's mine operating costs are from bids received in late 2017 for the open pit operations and from historical operations data updated to current costs for underground operations. The mill operating cost estimate is predicated upon historical September 2008 year-to-date operating costs and an update to reflect anticipated throughput and current costs. Both major cost centers were based upon local labour costs. It is assumed that the workforce levels are sufficient to support the planned increase in production rate and the open pit contract mining approach. Other than labour and assay laboratory costs, most other expenses were considered variable costs, thus increasing in direct proportion to the plant throughput and mining ratios.

Major capital costs for the restart of operations include equipment down payment; mine pre-stripping; increase of the reclamation bond; site utilities and services refurbishment; power system upgrades; administration building refurbishment; mine dewatering; process water upgrades; the installation of new primary crushing and stacking; secondary crusher with a refurbished tertiary crusher, replacement of the vibrating screen, new fine ore bin feeders; new reagent systems; onstream analysis; Zn thickener upgrades; replacement of the two concentrate vacuum disc filters and dryers with two vertical plate pressure filters; and basic plant automation for fine ore, grinding, and flotation.

The economics of the project are most sensitive to metal prices, the grade of the potentially-mineable mineralization, and exchange rate. The results of the sensitivity analysis are shown in Figure 1-1 (8% discount rate case), as related to the base case post-tax NPV of $107.7 million.

Figure 1-1: NPV8% Sensitivity*



* Metal Price (1) shows the Zn metal price sensitivity with the Pb price staying constant above US $1.15/lb and Metal Price (2) shows the Zn and Pb metal prices changing proportionally. Refer to Section 22.3.

1.2 Technical Summary

1.2.1 Property Description and Location

The Gays River Deposit ("the Property") is located approximately 60 km northeast of Halifax in the community of Gays River, within the Halifax Regional Municipality. The property's general location is 45°02' North, 63°21' West. The Gays River Deposit was divided into two zones: the Main Zone, south of Highway 224, and the Northeast Zone, which lies northeast of the highway and partly under Gays River. The Getty Deposit is located northwest of the Gays River Deposit on the western side of Gays River. The two deposits are separated by less than one kilometer. Access to the property is by paved roads and is approximately 15 km off the Trans-Canada Highway along Route 224. The Halifax International Airport is located 20 km southwest of the mine site.

The resources for the Getty deposit are included in this report. However, the deposit was excluded from the economic assessment in this study. It presents a significant mine expansion potential beginning in Year 8 and will be evaluated in the future.

1.2.2 Land Tenure

A Mineral Lease covers the Gays River Deposit. It consists of 648 hectares of mineral rights, including land with exploration potential for Zn / Pb mineralization, and 712.5 hectares of land ownership (real property) having surface rights. There are also seven exploration licenses in the general vicinity of the mine. All lands are in good standing and are registered to ScoZinc Limited.

Mineral Lease No. 10-1, which covers the entire mine site (Gays River Deposit), was originally granted by the Nova Scotia Government to Westminer Canada Limited on 02 April 1990. It was transferred to ScoZinc in 2002. The duration of the Mineral Lease is 20 years, at which time it may be renewed.

Regarding the Getty Deposit, Cullen et al. (2011) stated that "in September, 2006 the provincial government tendered exploration rights to the closed Getty property and Exploration Licenses 6959 and 6960 were subsequently issued to Acadian on October 20, 2006 as the successful bidder under the tendering process."

ScoZinc currently holds the mineral rights to the Gays River and Getty Deposits, as well as the mining rights and surface rights for Scotia Mine (ScoZinc Operations / Gays River Deposit). The existing surface rights are sufficient for currently planned mining operations.

1.2.3 History

The Gays River Deposit was discovered in 1973 by the Imperial Oil Enterprises ("Esso") / Cuvier Mines joint venture. Esso initiated development of an underground mine in 1978 and commissioned the mill in 1979. From 1979 to 1981, the mine produced 554,000 t of ore containing 2.1 % Zn and 1.4 % Pb. The mine closed in 1982 due to groundwater inflow and operating losses caused by low metal prices.

Seabright Resources Inc. (Seabright) acquired the mine and mill in 1984. Despite a favourable feasibility study, they did not reactivate the mine due to depressed metal prices at the time. They converted the mill for gold processing and processed gold ore from several satellite properties.

With the takeover of Seabright by Western Mining Corporation (Westminer) in 1988, a review of the potential for mining the deposit was undertaken. Following completion of feasibility studies in 1989, the underground workings were dewatered and test mining was carried out. A total of 187,000 t were mined over a 15 month period, with average grades of 7.47% Zn and 3.50% Pb. In 1991, production was suspended again due to groundwater inflow and economic considerations.

In 1997, Savage Resources Canada Limited (Savage) acquired the Scotia Mine assets from Westminer. Savage concluded that an open pit operation was feasible and initiated environmental permitting, including provisions for a diversion of a portion of the Gays River. Savage was subsequently taken over by Pasminco Resources Canada Company (Pasminco Resources) and their environmental assessment plan was approved by the Nova Scotia Minister of the Environment in August 2000.

Regal Mines Limited (Regal Mines) purchased Pasminco Resources which was later acquired by OntZinc in 2002. OntZinc later changed its name to HudBay Minerals Inc. (Hudbay). In 2006, Acadian Gold Corp (Acadian Gold) purchased 100% of ScoZinc and all of its assets (consisting mainly of Scotia Mine and its infrastructure) from OntZinc for $7 million.

ScoZinc reactivated the mill and surface-mined the Gays River Deposit during 2007 and 2008. Depressed metal prices forced ScoZinc to place the mine on care-and-maintenance status at the end of 2008. In May 2011, Selwyn Resources Ltd. (Selwyn) purchased ScoZinc Limited and all of its assets, including the Scotia Mine and ScoZinc's exploration claims, for $10 million less a deduction relating to increased reclamation bonding requirements that were being determined at the time of the acquisition and outstanding mineral royalty taxes due to the Nova Scotia government.

1.2.4 Geology

The Property is underlain by basement rocks of the Cambro-Ordovician Meguma Group, which had significant local topographic relief due to rift faulting and erosion. Locally, a veneer of Horton Group, red-brown conglomerate, and sandstone mark the base of the unconformably overlying Lower Carboniferous rocks, which host the Gays River Deposit. In areas where the basement rocks formed islands in the Carboniferous Sea, coral reefs formed along the shores. These carbonate rocks are the Gays River Formation. The MacCumber Formation is time-equivalent to the Gays River Formation. The MacCumber and Gays River Formations are overlain by evaporites of the Carroll's Corner and Stewiacke Formations.

The Gays River Formation mineralization has long been considered a Mississippi-Valley-type Pb-Zn deposit. This type of deposit is carbonate-hosted, classified as a typical open space filling type, and hosted in a dolomitized limestone. The limestone developed as a carbonate build-up on an irregular pre-Carboniferous basement topographic high where conditions allowed for growth of reef-building organisms.

The Zn- / Pb-bearing Gays River Formation trends in an east-northeast direction across the Property. Locally, the mineralization dips 45° on average, and up to vertical in places, to the north-northwest which is the depositional slope of the front of the Gays River reef unit. But, the dip tends to be horizontal in the back reef area (south of the main trend). The mineralization is present as sphalerite and galena and grades from massive Pb-Zn ore-grade material in the fore reef to finely disseminated, lower grade material in the back reef. In the mine area, the Gays River Formation is overlain either by the evaporites of the Carroll's Corner Formation and/or overburden.

1.2.5 Mineral Resources

Only Mineral Resources were identified. As this is a PEA, there are no Mineral Reserves.

1.2.5.1 Gays River Deposit Resource Estimate

As detailed in the recent ScoZinc resource technical report ("Updated Mineral Resource Report for the Gays River and Getty Deposits", 8 October 2012), an updated mineral resource estimate was completed in 2012 based on verified sampling results and confirmed that the sample types and densities were adequate for establishing Mineral Resources. The sampling results were representative of the mineralisation. The available information and sample density allowed a reliable estimate to be made of the size, tonnage and grade of the mineralisation in accordance with the level of confidence established by the Mineral Resource categories in the CIM Standards.

For Mineral Resource calculation, the Gays River Deposit was divided into two zones: the Main Zone, south of Highway 224 and the Northeast Zone, which lies northeast of the highway and partly under Gays River. For both zones, manual interpretation was required to properly model the geology. The Main Zone was broken down into a high-grade (HG) mineralized zone and a low-grade (LG) mineralized zone. Drill-hole data and underground openings were then plotted on hard-copy plans at ten metre intervals, and interpretations of the high-grade zone, the low-grade zone and the hanging-wall 'Trench' were produced.

The non-diluted mineral resources in the Gays River Deposit using a 0.75 % zinc-equivalent cut-off are presented in Table 1-1.

Table 1-1: Gays River Deposit Mineral Resources

Resource Category	Zn Eq. % Cut-off	Tonnes (Rounded)	Zinc %	Lead %	Zinc Eq %[1]
Measured*	0.75	2,075,000	3.14	1.68	5.16
Indicated*	0.75	5,770,000	3.30	1.69	5.32
Indicated + Measured*	0.75	7,845,000	3.25	1.69	5.28
Inferred*	0.75	3,677,000	2.35	1.51	4.16

* Denotes Base Case for this study. Refer to table 14-1 for resource estimation notes.

[1] Zinc Equivalent % (Zn Eq.%) = Zn % + (Pb % x 1.18) and is based on mill recoveries of 89.3% for zinc and 89.5% for lead, $US1.10/lb Zn and $US1.15/lb Pb metal pricing and smelter returns of 85% for Zn and 95% for Pb.

The majority of the outlined mineral resources could likely be mined using surface mining methods. Some of the identified mineral resources are located underneath Gays River. Sandy soil lies underneath Gays River, so mining close to the river would be susceptible to water inundation. In other words, the mineral resources that lie close to, or underneath Gays River would be relatively more expensive to recover due to the added cost of either (a) diverting the river or (b) recovering the higher-grade portions of the mineral resources using underground mining methods.

1.2.6 Mining Methods

The two conventional open pits and the proposed underground mine will provide a blended feed to the mill. Production scheduling is based on an average production rate of 852,800 tonnes per year (or 2,600 tonnes per day) into the mill over an average of 328 operating days per year. The average waste to ore ratio for the LOM open pits is 12.0 to 1 (excluding pre-stripping which is included in the capital costs). Approximately 63% of the waste is readily removed without blasting, including soils that will be used for reclamation, and 22% of the waste is gypsum, which will be stockpiled for possible future sale: no value for gypsum has been used in the PEA. Open pit mine dilution and mining losses are assumed to be 7.5% and 5%, respectively. The material movement rate, including ore and waste, in the 7-year production schedule peaks at approximately 57,000 tpd. In-pit diluted mineral resources are 6,374,839 tonnes grading 3.00% zinc and 1.50% lead.

The underground operation is based on Cut-and-Fill mining with un-cemented backfill, producing up to 500 tonnes per day with an average of 294 tonnes per day of higher-grade mill feed. A drawdown of the water table in the proposed mine area would be achieved largely by the pumping associated with the open pit operations supplemented with wells as appropriate. The development of the underground mine access requires a sustaining capital investment of about $11.7 million, most within Year 5 of the overall mining schedule, to develop the access to the higher-grade zones. Diluted and recoverable underground mineral resources are estimated at 289,300 tonnes grading 6.54% zinc and 3.74% lead. This material will be blended with open pit and stockpile feed to the mill over approximately two years beginning in the second half of Year 5 of the Life of Mine plan.

Aggregate production from the two open pits and the underground mine is estimated at 6,552,000 tonnes grading 3.06% zinc and 1.57% lead.

1.2.7 Mineral Processing and Metallurgical Testing

This report proposes to raise the mill throughput to a nominal 2,600 tonnes per day or 852,800 dmt per annum, by effecting changes to the crushing, grinding and concentrate filtration circuits.

The following items were identified as contributors to the poor plant reliability and performance:

Mining

- In order to achieve production targets dewatering of the pit will be a critical success factor. The plan addresses concerns related to the volume of ingress water as well as the quality of water being removed.
- Establishing sufficient mill feed stockpiles to support steady state production and allow for blending to better manage grade control.

Crushing and Grinding

- The size of run of mine ore varied dramatically and with limited primary crushing circuit capacity the crushing plant throughput was reduced or restricted. This is addressed with introduction of a new primary jaw crusher and stockpiles outside of the mill.
- The ore entry point into the mill was open to the elements resulting in wet feed and freezing conditions on a regular basis. A covered entry point is proposed.
- The crushing circuit capacity was limited as a single deck screen 6’ × 12’ and a single secondary crusher struggled in terms of product sizing and overall capacity. A new 6’ × 20’ double deck screen integrated with secondary and tertiary crushing.
- The fine ore bin discharge used slot feeders that were neither controlled nor easily maintainable. New feeders and automation will be introduced to improve bin discharge and flow.
- There was insufficient process water available for Rod and Ball mills at higher throughputs. The reclaim water / process water system will have its capacity increased to match the desired 2,600 tpd.

- The grinding circuit was manually controlled by the operator. Automation will be introduced in the grinding circuit.

Flotation

- There was no feedback to the operators on critical process streams assays. The fully manual and visual control will be replaced with reagent controls, pH monitoring, flotation cell level control and on stream analysis.

Dewatering and Filtration

- For both the Pb and Zn thickeners the feed well did not achieve the design intent to evenly distribute feed, provide for de-aeration and reduce turbulence facilitating better solids settlement. These will be replaced.
- The Zn thickener rakes were manually controlled and with changes in thickener loading, product density varied. New automated rakes will be installed.
- The disc filters and oil fired dryers were difficult to maintain and expensive to operate. These will be replaced with plate filters.

The projected metallurgical performance provides for a Pb concentrate grading 71% Pb at 85.7% recovery, and a Zn concentrate grading 57% Zn at 86% recovery. The operation is expected to mirror previous operational performance as the plant undergoes significant upgrades and operational improvements.

Capital costs are included for modernizing the crushing, grinding, flotation, and dewatering processes as well as for improved instrumentation.

No deleterious minor elements are expected in the concentrates. The concentrates should be readily marketable, given their clean high-grade nature.

1.2.8 Project Infrastructure

The Scotia Mine mill, designed and built in 1978/1979, is a flotation process and had an initial rated capacity of 1,350 tonnes per day. However, it has operated for extended periods at a rate in excess of 2,200 tonnes per day. Most infrastructure required for mineral extraction and processing is available onsite.

Highway access to the site is excellent. The road network and other civil infrastructure is in good condition with typical minor maintenance being required. Before production may occur, integrated with the pre-stripping operation, roadwork will be completed onsite to service the expanded production area.

Storage and ship loading facilities for lead and zinc concentrates are available at the seaport of Sheet Harbour, a distance of 80 km from the mine site over paved roads. ScoZinc leases land and infrastructure from the Province and owns the conveyors and ship loader. Rail transport facilities have also been used for concentrate shipping via the port in Halifax. A railway siding is located in Milford, 8 km from the site on paved roads.

Three-phase power is supplied through the regional grid at reasonable rates. Most of the mill's water requirements are satisfied by in-process recycling. Make-up water is drawn from the perennial Gays River. The existing tailings pond has sufficient capacity for the life of the project. There is also sufficient area for waste rock storage on the property.

1.2.9 Environmental

The ScoZinc Mine is an existing operation with substantive environmental databases, operating history, and valid permits and licenses that allow for the mining, processing of ores, and the shipping of concentrates. The site has operated several times in the past as a fully permitted underground and, more recently, surface mine. The most recent operations by ScoZinc were completed under the Environmental Assessment (EA) approval granted in 1999 to Savage Resources and transferred to ScoZinc. The Industrial Approval (IA) and other minor operating approvals needed (Water Withdrawal and Septic System Operation for example) were in place during the previous operations and transfers are complete. The majority of the resources used in this economic analysis are already under permit and mining of these resources (the Main pit and Southwest Expansion) may begin with the completion of any residual Industrial Approval conditions.

Another important aspect of the project status with respect to permits, environment and community is the experience of regulators and community with the project and the fact that environmental baseline conditions are already understood. In combination, these factors reduce the overall permitting risk and anticipated timelines for permitting of project expansions to include the entire mineral resource used in this analysis.

In addition, the risks and potential costs associated with environmental and community issues are well understood and based on operating experience and history of the mine. As such the financials for environment and community matters that are input to the economic model are reasonable assumptions.

2.0 INTRODUCTION

This updated PEA was prepared for the ScoZinc Property, containing the Gays River deposit, located in central Nova Scotia, Canada, by Stantec Consulting Ltd. (Stantec) in conjunction with ScoZinc Limited (ScoZinc).

The current and previous resource estimates were prepared and disclosed as required under National Instrument 43-101 (NI 43-101) and are considered compliant with Canadian Institute of Mining, Metallurgy and Petroleum Standards for Mineral Resources and Reserves. Selwyn Resources Limited (Selwyn) updated mineral resources in 2012 (see 24 August 2012 news release), following a 2011 drill program, reanalysis of historical data, and the 2012 remodeling of the resource. This resulted in a 55% and 65% increase of Measured and Indicated Mineral Resources, respectively, as compared with the prior Mineral Resource inventory (06 April 2011 news release). The expanded Mineral Resource formed the basis for a revised mine plan and economic model (22 November 2012 news release). That revised mine plan confirmed a significant increase in mine life for the Main and Northeast pits.

This update to the PEA builds on that mine plan and incorporates a proposed underground mining operation between the Main and Northeast open pits, and blending of the high-grade material with the lower grade open pit mineralization in Years 5 to 7 of the mine plan.

A re-start capital plan is included in the update along with a change to the mining approach to include contract mining for all open pit activities. Capital and operating costs have been updated to reflect current costs and market conditions.

The capital re-start plan has addressed known deficiencies and limitations based on past operating experience.

An examination of metallurgical throughput and performance in the 2008 operating period, relying on data provided by ScoZinc, was used as a basis for this update.

The detailed economic assessment is classified as a PEA due to the fact that the mine plan includes a small proportion of Inferred mineral resources.

2.1 Extent of Field Involvement of the Qualified Person(s)

Field and site visits by Mike Romaniuk, Jason Baker, and Steve Oaks took place in late 2017 and early 2018, and were conducted with extensive consultation with ScoZinc current and past employees. During those visits, the mine property and mill facilities were viewed, and Steve Oaks included a visit to the Sheet Harbour facilities.

3.0 RELIANCE ON OTHER EXPERTS

This report was prepared by Stantec. The material, conclusions and recommendations contained herein are based upon information available and supplied by ScoZinc to Stantec at the time of report preparation, as well as independent reviews

Stantec consulted ScoZinc during the writing of this report. Stantec has no reason to question the quality or validity of the data and opinions expressed by these experts. Stantec supports the data and conclusions of those Qualified Persons who have been included in this report.

This report includes opinions that concern exploration and development potential for the project as well as recommendations for further analysis. These are intended to serve as guidance and should not be taken as a guarantee of success.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Gays River (Main and Northeast) Deposit is located at approximately 45°02′ North, 63°21′ West, 60 km northeast of Halifax, Nova Scotia, in the community of Gays River in the Halifax Regional Municipality.

The Gays River Deposit consists of 648 ha of mineral rights in the form of three contiguous mineral leases, including land with exploration potential for zinc (Zn) / lead (Pb) mineralization, and 712.5 ha of land ownership (real property) (see Figure 4-1 and Figure 4-2).

The Getty Deposit consists of 16 contiguous mineral claims, approximately 259 ha.

Figure 4-1: Location Map

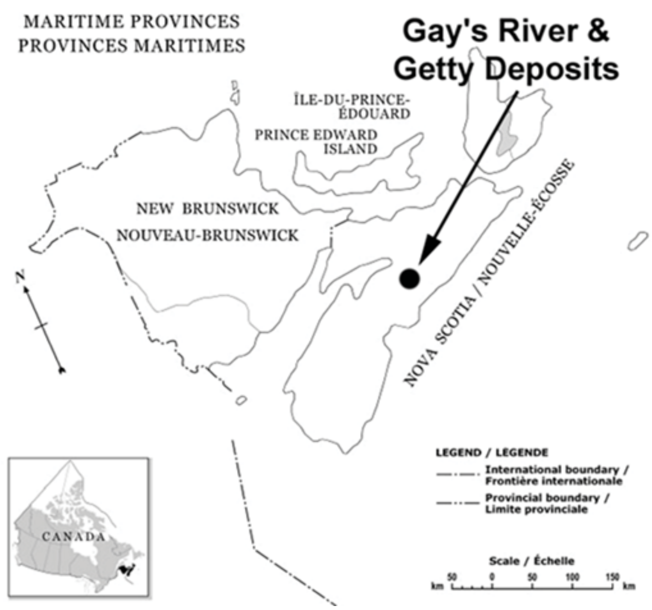
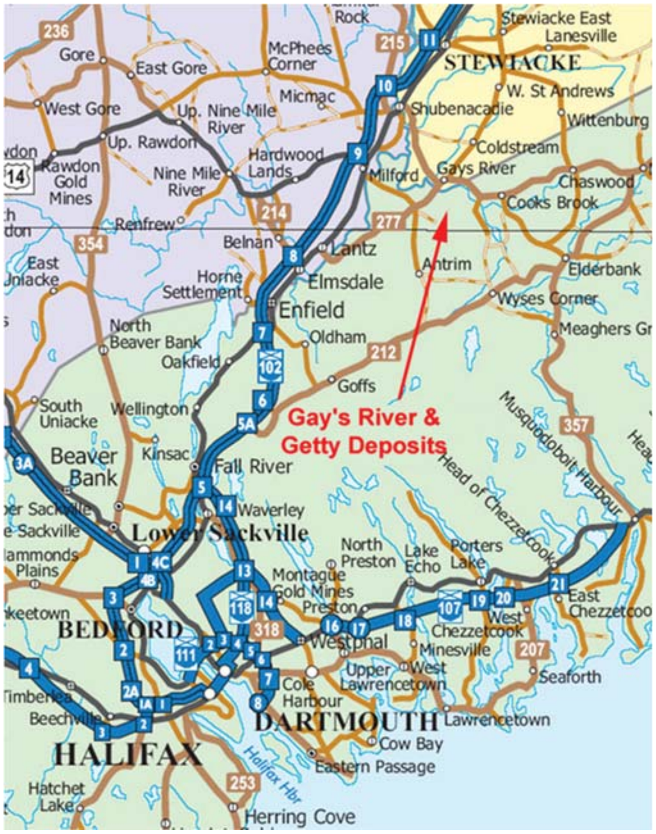


Figure 4-2: Location Relative to Halifax



4.1 Exploration Licenses

ScoZinc currently controls 5 exploration licenses, covering 41 claims in the vicinity of the mineral leases (see Figure 4-3). Each individual claim covers an area of approximately 16.2 ha (40 acres), for a total of approximately 664 ha (1,640 acres). These licenses are located along strike from the Gays River Deposit and include favourable host rocks similar to that at the mine site.

All exploration licenses were in good standing or pending approval of renewal application and registered to ScoZinc Limited as of 02 January 2018. The ScoZinc exploration licenses are summarized in Table 4-1. Table 4-2 through Table 4-6 provide additional details on each of the licenses.

Figure 4-3: Claim Reference Map Showing Exploration Licenses, Mineral Lease, and Real Property Boundary (Surface Rights) for the Gays River and Getty Deposits

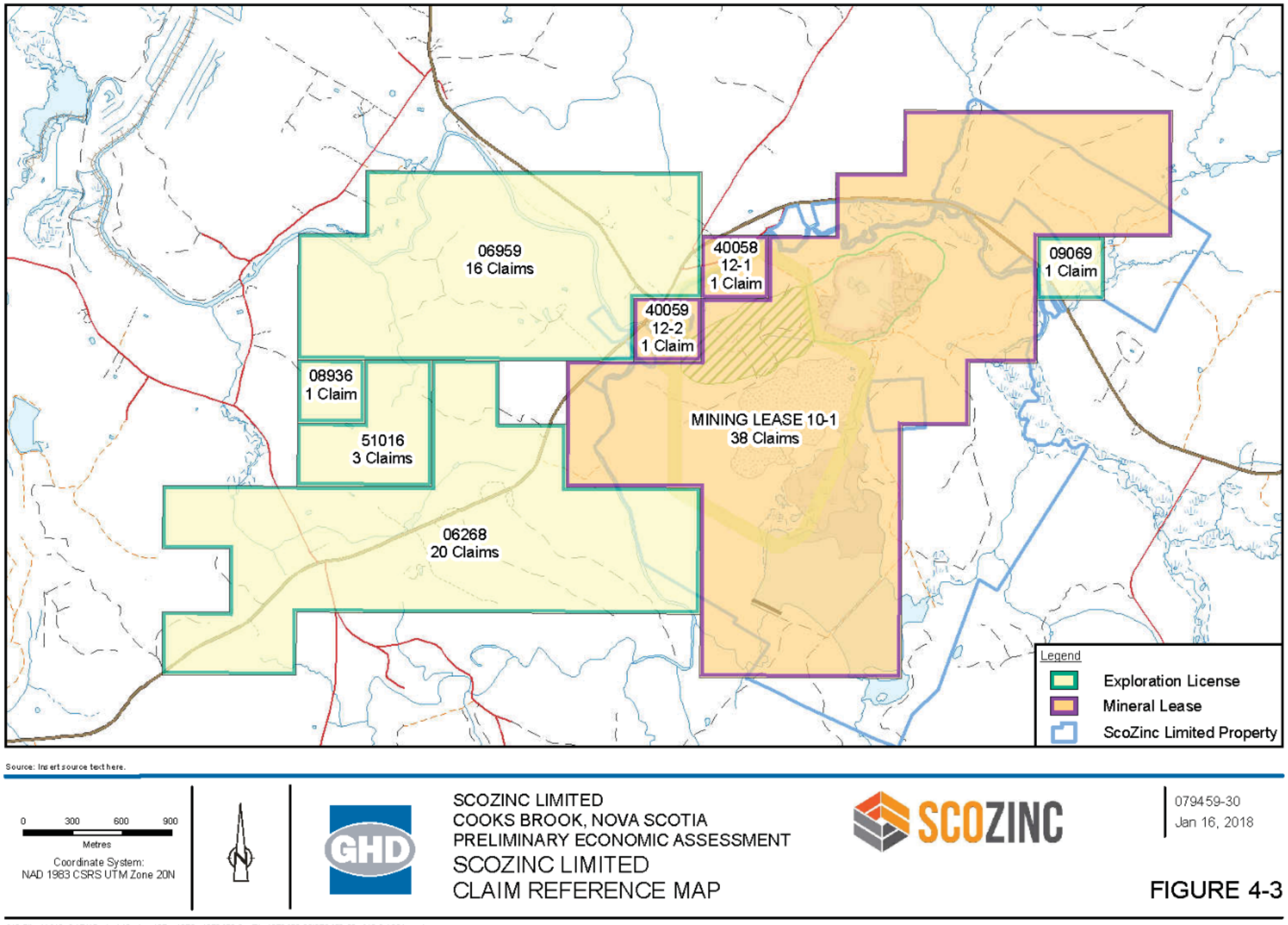


Table 4-1: Summary of ScoZinc Exploration Licenses

License	No. of Claims	Sheet	Anniversary Date	Year of Issue
06268	20	11E/03B	02 May 18	21
06959	(Getty) 16	11E/03B	20 Oct 18	12
08936	1	11E/03B	21 Dec 18	8
09069	1	11E/03B	19 Aug 18	13
51016	3	11E/03B	05 May 18	2

Table 4-2: Exploration License 06268 (20 Claims)

Claim Reference Map	Tract	Claims	Anniversary Date
11E/03B	7	NO	02 May 18
	18	ABC EFGH	
	19	ABCD EFGH LM N	

Table 4-3: Exploration License 06959 (Getty Deposit, 16 Claims)

Claim Reference Map	Tract	Claims	Anniversary Date
11E/03B	30	BCD EFGH JKLM	20 Oct 18
	31	ABGHJ	

Table 4-4: Exploration License 08936 (1 Claim)

Claim Reference Map	Tract	Claims	Anniversary Date
11E/03B	18	P	21 Dec 18

Table 4-5: Exploration License 09069 (1 Claim)

Claim Reference Map	Tract	Claims	Anniversary Date
11E/03B	28	F	19 Aug 18

Table 4-6: Exploration License 51016 (3 Claims)

Claim Reference Map	Tract	Claims	Anniversary Date
11E/03B	18	JKQ	05 May 18

4.2 Mineral Lease

Three contiguous mineral leases (10-1, 40058, and 40059) cover the Scotia Mine site (Gays River Deposit). The anniversary date (review date) of Mineral Lease 10-1 is 02 April of each year. The anniversary date of Mineral Leases 40058 and 40059 is 02 October of each year. Table 4-7, Table 4-8, and Table 4-9 list the claims comprising the mineral leases. Figure 4-3 and Figure 4-5 show their locations.

Note: Mineral Lease 10-1 was originally granted by the Nova Scotia Government to Westminer Canada Limited on 02 April 1990 as a "mining lease." However, changes to the Nova Scotia Mineral Resources Act that came into effect in November 2004 changed the terminology such that existing "mining leases" are now known as "mineral leases."

The leases convey the rights to all minerals except coal, uranium, salt, and potash. The leases were then transferred to Savage Resources in 1996, and later to Pasminco Resources Canada Company in 1999. It was finally transferred to ScoZinc in 2002. The duration of the lease is twenty years, at which time it may be renewed. The expiry date of the lease is 02 April 2030.

The Nova Scotia government currently holds a reclamation security (bond) for the three leases in the amount of approximately $2.77 million.

In addition, as a condition of the acquisition of ScoZinc Limited in 2011, Selwyn instructed its Nova Scotia counsel to pay the Nova Scotia government $892,876.72 in provincial royalty payments for ScoZinc's past production.

Table 4-7: Mineral Lease 10-1 (38 Claims), Tract Map (NTS) 11E-03B

Tract	Claims	Number of Claims
5	NOP	3
19	JKPQ	4
20	BCDE FGK LMNO PQ	13
28	DEKL MNOP	8
29	ABCD FGH JKQ	10
Total		**38**

Table 4-8: Mineral Lease 40058 (1 Claim), Tract Map (NTS) 11E-03B

Tract	Claims	Number of Claims
29	E	1
Total		**1**

Table 4-9: Mineral Lease 40059 (1 Claim), Tract Map (NTS) 11E-03B

Tract	Claims	Number of Claims
30	A	1
Total		**1**

4.3 Surface Rights (Real Property)

4.3.1 Gays River Deposit

ScoZinc owns outright approximately 1760.37 acres of land (real property) containing the entire surface infrastructure, the tailings area, and most of the outlined mineralization (refer to Table 4-10 and Figure 4-4). The boundaries were established through legal surveys.

Table 4-10: Property Ownership, ScoZinc Limited

PID	Filename	Update Date	Area (ha)	Area (ac)	Corporation Name
40757577.00	mu0867	20110311.00	71.20	175.85	ScoZinc Limited
20080495.00	mu0867	20110311.00	3.70	9.05	ScoZinc Limited
20223418.00	mu0867	20110311.00	0.30	0.83	ScoZinc Limited
40227951.00	mu0867	20110311.00	46.40	114.70	ScoZinc Limited
41239542.00	mu0867	20110311.00	0.00	0.04	ScoZinc Limited
40290264.00	mu0867	20110311.00	43.50	107.49	ScoZinc Limited
00369363	mu0867	20110311.00	15.20	37.60	ScoZinc Limited
40227969.00	mu0867	20110311.00	2.40	5.89	ScoZinc Limited
40312092.00	mu0867	20110311.00	9.40	23.25	ScoZinc Limited
40227985.00	mu0867	20110311.00	0.40	0.87	ScoZinc Limited
40291452.00	mu0867	20110311.00	222.30	549.35	ScoZinc Limited
00373423	mu0867	20110311.00	2.30	5.63	ScoZinc Limited
40290256.00	mu0867	20110311.00	58.80	145.19	ScoZinc Limited
00522201	mu0867	20110311.00	35.60	87.85	ScoZinc Limited
41358136.00	mu0867	20120224.00	2.90	7.10	ScoZinc Limited
41283268.00	mu0867	20110311.00	11.20	27.57	ScoZinc Limited
00373621	mu0867	20110311.00	41.50	102.44	ScoZinc Limited
40746786.00	mu0867	20110311.00	23.80	58.72	ScoZinc Limited
00522623	mu0867	20110311.00	37.00	91.38	ScoZinc Limited
40763872.00	mu0867	20110311.00	13.80	34.08	ScoZinc Limited
41094400.00	mu0867	20110311.00	33.00	81.63	ScoZinc Limited
41358128.00	mu0867	20110311.00	0.70	1.76	ScoZinc Limited
20080495.00	mu0415	20110225.00	19.90	49.24	ScoZinc Limited
20080529.00	mu0415	20110225.00	3.40	8.51	ScoZinc Limited
20313250.00	mu0415	20110225.00	0.60	1.60	ScoZinc Limited
20080511.00	mu0415	20110225.00	3.30	8.08	ScoZinc Limited
20158184.00	mu0415	20110225.00	2.50	6.14	ScoZinc Limited
20223418.00	mu0415	20110225.00	1.70	4.28	ScoZinc Limited
20416384.00	mu0415	20110225.00	1.20	3.08	ScoZinc Limited
20158176.00	mu0415	20110225.00	2.40	5.96	ScoZinc Limited
40757577.00	mu0415	20110225.00	2.10	5.21	ScoZinc Limited
Total			**712.5**	**1,760.37**	

Figure 4-4: ScoZinc Property Map as of January 2018

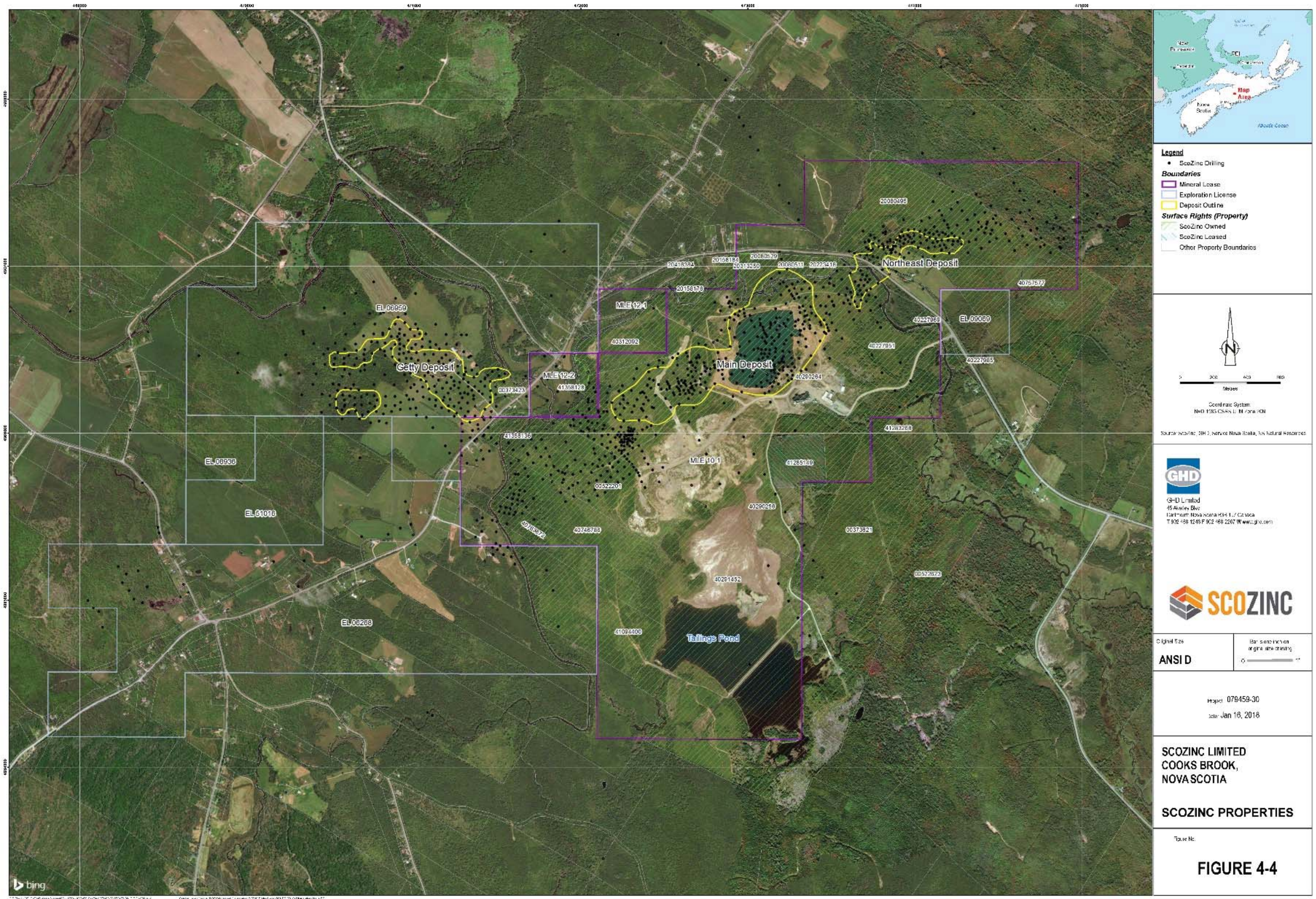
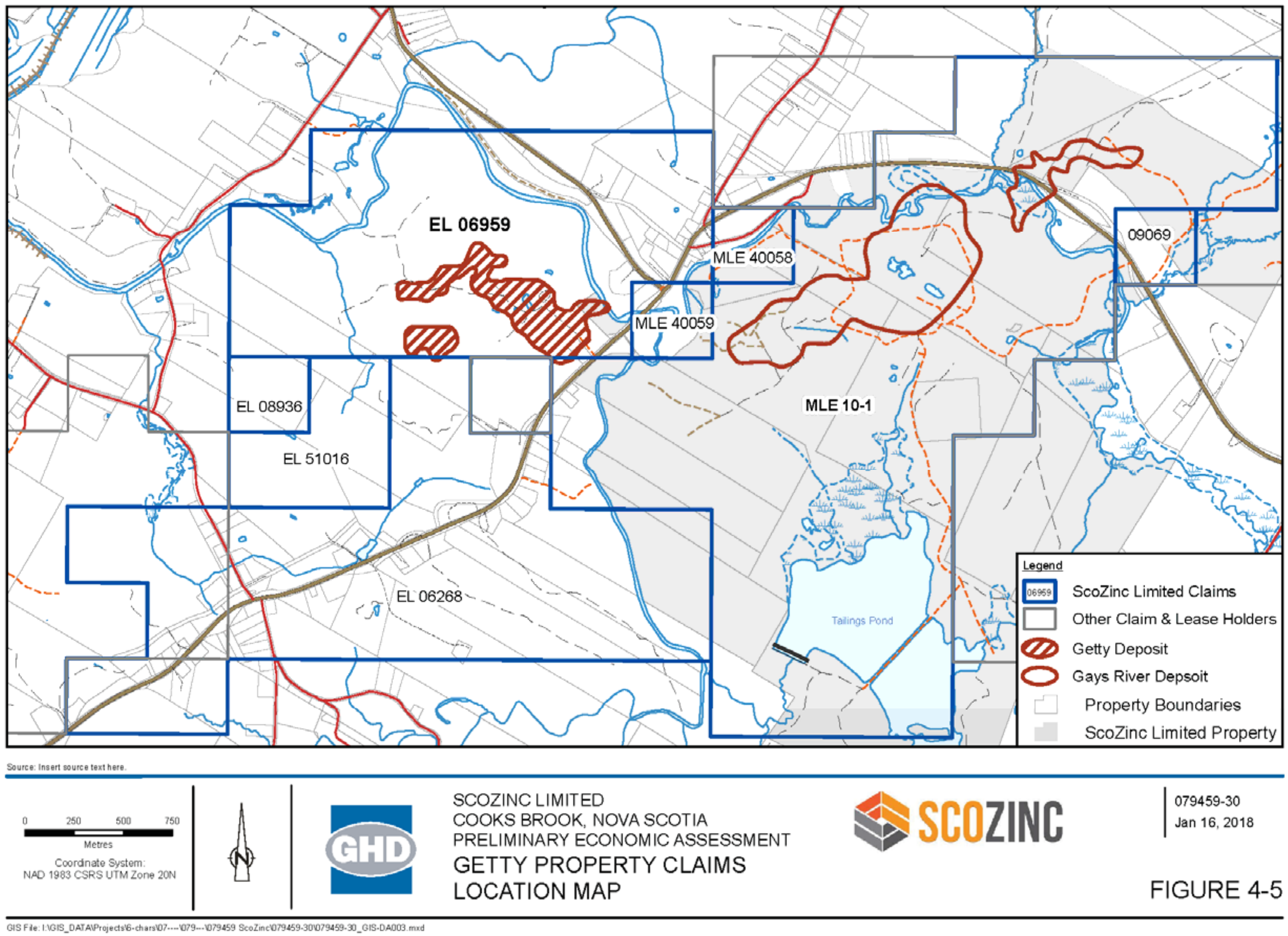


Figure 4-5: Claim Reference Map for the Getty Deposit



4.4 Aggregate Lease

An aggregate lease covers the Scotia Mine property. Gallant Aggregates (Gallant) signed a 30-year lease agreement to mine and remove aggregate from the property for $1.00/t of material that is removed from the property. The lease was signed on 15 May 2003 and entitled Gallant, with certain limitations, to mine anywhere on ScoZinc's land. The agreement contains a renewal clause and gives Gallant the right of first refusal to purchase the surface rights (real property titles). A major condition of Gallant's lease is that metal mining takes precedence over aggregate mining. Therefore, Gallant's lease would not interfere with Zn and Pb mining operations.

In January 2008, Gallant exercised its option under the agreement to purchase approximately 25 acres of the Scotia Mine property. Concurrent with the transfer of the 25 acres, ScoZinc and Gallant executed a License, Option, and Royalty Agreement, which terminated the original agreement and granted Gallant the right to access the Scotia Mine property to access existing water infrastructure and to obtain electrical power. The License, Option, and Royalty Agreement grants Gallant the right to remove, extract, and process sand, gravel, and fill, and obtain materials from the overburden and waste material created by ScoZinc at the Scotia Mine property for the greater of $25,000/a or $1.00/t. In addition, Gallant has the right of first refusal to purchase the Scotia Mine property if ScoZinc plans to sell the property after mining operations are completed or abandoned.

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

5.1 Accessibility

The Gays River (Main and Northeast) and Getty Deposits ("the Property") are located approximately 55 km northeast of Halifax, Nova Scotia, along the border between Colchester and Halifax Counties (45º01'55" North Latitude and 63º21'30" West Longitude). It lies approximately one kilometer east of the community of Gays River. Access to the property is by paved roads and is approximately 15 km off the Trans-Canada Highway, along Route No. 224. The Halifax International Airport is located 20 km southwest of the mine site.

5.2 Climate

The temperate climate permits year-round operations.

The following climate information reported for Upper Stewiacke during the 30-year period ending in 2010 characterizes seasonal precipitation and temperature trends in the area. The average July daily mean temperature for the reporting period is 18.4 °C with a corresponding average maximum daily temperature of 23.6 °C. Average daily winter temperature for January is –6.8 °C with a corresponding average daily minimum being –12.0 °C. Mean annual temperature is 6.2 °C, and mean annual precipitation is 1,363.6 mm. Climate conditions permit many exploration activities, such as core drilling and geophysics, to be efficiently carried out on a year-round basis. Other activities, such as geochemical surveys and geological mapping are typically limited by winter snow cover. (Government of Canada 2017).

5.3 Local Resources and Site Infrastructure

The Scotia Mine mill, designed and built in 1978/1979 had a nominal ("nameplate") capacity of 1,350 tonnes per day (t/d) (see Figure 5-1). However, during 2007-2008, ScoZinc operated the mill for extended periods at rates over 2,200 t/d. It was initially built to treat the Zn / Pb ore from the Gays River Mine. In 1986, it was modified to treat gold ores using gravity and flotation circuits. In 1989, it was reworked again to treat Zn / Pb ore from the Scotia Mine, which was then being operated by Westminer Canada Ltd. (WMC). The concentrator has been maintained and is ready for reconditioning and restart after a modest capital expenditure. A number of processing restrictions have been identified which limited availability and throughput. and these are addressed as part of the re-start capital program.

Figure 5-1: Site Infrastructure (Facing Southwest)

The mill is equipped with three stage crushing, two stage grinding, flotation cells, thickening, and plate filter dewatering. The concentrator building contains a complete analytical laboratory, metallurgical testing laboratory, control room, maintenance area, and office facilities. Its total area is approximately 32,000 ft^2.

The administration building has an area of approximately 26,000 ft^2. It contains offices, a dry, a warehouse, workshops, a large boardroom, and several heavy equipment bays. Other, smaller surface facilities include:

- A potable water pumping system from the Gays river;
- A potable and fire water storage, filtration, and pumping building;
- A pit dewatering pumping and piping system including associated electrical systems;
- A reclaim water storage area with controlled and emergency water release infrastructure;
- A reclaim water system that provides process water for milling operations;
- A waste rock storage area;
- A tailing storage facility with associated dams and decant systems;
- Power lines and transformers necessary for dewatering, administration, and mill operations;
- A welding shop;
- A geology building; and,
- A core shed.

Storage and ship loading facilities for Pb and Zn concentrates are available at the seaport of Sheet Harbour, a distance of 80 km from the mine site over paved roads. ScoZinc owns loading equipment and a storage facility on lease land at the Sheet Harbour Marine Industrial Park. The Sheet Harbour lease expires in April 2018 and has a 10-year renewal option. Efforts are underway to renew the lease.

Sheet Harbour is a natural harbour on the Atlantic coast that remains ice-free in the winter months and can handle vessels up to 40,000 t in displacement. Rail transport facilities have also been used for concentrate shipping. A railway siding is located in Milford, 8 km from the site.

During the last period of operations, Pb concentrate was shipped through the port of Halifax, approximately 70 km from the mine over excellent roads. Zn concentrate was shipped in bulk through port facilities at Sheet Harbour that ScoZinc leases.

The existing surface rights are sufficient for mining operations. Power is supplied through the regional grid at reasonable, industrial rates. Scotia Mine owns and maintains step-down transformers adjacent to the mill. Most of the mill's water requirements are satisfied by in-process recycling. Make-up water is drawn from the perennial Gays River.

The existing tailings pond is large enough for the life of the proposed operation. It is located just south of the mill on the footwall side of the deposit. Its design capacity was 10 million tonnes. Approximately two million tonnes of tailings have been stored there, leaving a current capacity of over eight million tonnes.

There is sufficient area for waste rock and overburden storage on the property. The main area for waste rock storage lies adjacent to the tailings pond on its northwest shore, on the footwall side of the deposit.

The site is in close proximity to Halifax and the surrounding population centers. This provides access to resources and services to support operations. In addition, there are a number of active mining and pit operations in the region providing access to both skilled personnel and services to support operations.

5.4 Physiography

The property is in a rural-residential area of central Nova Scotia that is typified by rolling topography and abundant surface water. The Gays River Deposit lies along the south side of the Gays River main branch, immediately east of the confluence with the Gays River south branch. The Getty Deposit lies immediately west of the Gays River Deposit, on the north side of Highway 224 (refer to Figure 4-5: Claim Reference Map for the Getty Deposit.)

The Gays River watershed is characterized by gently rolling topography, having a maximum elevation of 170 m, an extensive cover of deciduous forest, a small population, and local agricultural land development. Lakes, ponds, and rivers are sparsely distributed throughout the watershed. Typical vegetation consists of northern black spruce, balsam fir, and juniper, with birch in more wet areas. Areas of open bog occur on part of the claims. Currently, parts of the forest are being harvested or thinned.

6.0 HISTORY

6.1 Overview

The Gays River Formation has seen exploration since the 19th century. Modern exploration on the Gays River Formation began in the early 1970s. From Cullen et al. (2011);

"First reports of zinc-lead mineralization in the Gays River area date to the late 1800's and from this time until the 1950's exploration consisted of limited amounts of mapping, pitting, trenching and sampling with up to 3% lead values being reported. Most activities focused on the area immediately around the adjacent Scotia Mine site, particularly along the South Gays River, where outcropping Gays River Formation dolomite hosting low grade zinc and lead mineralization was trenched and drilled in the 1950's in the "Gays River Lead Mines Area" (Campbell, 1952)." (Cullen et al., 2011, section 5.2).

6.1.1 Gays River Deposit

The history of the project begins with its discovery in the early 1970s by Cuvier Mines Limited (Cuvier). Cuvier and Imperial Oil Limited (Esso) carried out exploration work and delineated the mineralized zone, which was then identified as being 4 km long, 220 m wide, with depths varying from 20 m to 200 m. Initial development consisted of an exploration decline driven in 1975/76 with mine development starting in 1978 and mill commissioning in October 1979.

From 1979 until 1981, Esso operated the mine and targeted the lower grade ore using a lower cost, bulk Room and Pillar mining method approach. Though Esso carried out some test mining in the higher-grade mineralization near the carbonate contact, it was not part of the mine plan at that time. During this period, 554,000 t of Zn / Pb ore was mined with an average grade of 2.12% Zn and 1.36% Pb (Table 6-1). Due to low metal prices, problems caused by high rates of water influx and difficult ground conditions, mining was suspended in 1981 and the mine was allowed to flood.

Table 6-1: Historical Milling Records

	Mill Feed			Concentrate Produced				Metal Recovery (%)	
	Tonnes	% Pb	% Zn	Pb Tonnes	Zn Tonnes	% Pb	% Zn	% Pb	% Zn
Esso (1979-1981)	550,000	1.40	2.10	10,000	17,000	73.6	61.5	95.6	90.5
WMC (1989-1991)	190,000	3.50	7.50	8,000	21,000	75.6	61.2	90.9	90.2
ScoZinc, 2007	337,000	0.85	2.14	3,359	8,694	64.4	55.4	75.5	66.7
ScoZinc, 2008	718,271	1.02	2.70	8,535	27,729	70.1	55.9	81.6	79.9
Total	1,795,271	1.00	2.92	29,894	74,423	72.1	58.6	87.8	83.2

In 1985, Seabright Resources Inc. (Seabright) purchased the property and modified the mill circuits to treat gold ore from other Nova Scotian properties.

In 1988, Westminer Canada Limited (WMC) purchased Seabright Resources. WMC began dewatering the underground mine in 1989. Their extraction method was to use narrow vein, Cut-and-Fill mining to extract the higher-grade ore zones. The mine was placed back into operation and reached commercial production in March 1990 (Figure 6-1 and Figure 6-2). During the period of operations by WMC (August 1989 to May 1991), the mine produced 190,000 t of ore at an average grade of 7.5% Zn and 3.5% Pb. Mining was curtailed due to low metal prices, mining method problems, and high rates of water influx. Also, for corporate reasons, WMC decided to focus on larger scale mining ventures. Following suspension of mining at Gays River Mine, WMC commissioned several studies to characterize the local hydrology of the mine and to control the ground water in the mine. These results were never tested during mining, since a cyclic low in metal prices, among other factors, prompted WMC to place the property up for sale.

Figure 6-1: Decline and Portal Access to the Underground Workings (circa 1990)

The background of this photo, where the equipment is working, was surface-mined by ScoZinc during 2007/2008.

Figure 6-2: Flotation Circuit (circa 1990)

In late 1996, Savage Zinc, Inc. purchased the Gays River Mine property from WMC and formed a wholly owned subsidiary named Savage Resources Canada Company (Savage). Savage started to rehabilitate the property, shops, equipment, and office, with the aim of starting production in 1997.

When Savage took over the operation of the former Gays River mining facility, the underground workings were flooded to the surface. After purchasing equipment and hiring employees, the mine dewatering phase started on 07 June 1997. With an installed pumping capacity of 9,000 usgpm, the average pumping rate to reach the 425 m Level was 5,200 usgpm. This level was reached during late August 1997. During this period of dewatering, men and equipment went underground to clean out the workings while management carefully examined the ground conditions. They decided to prepare a mine plan that considered an open pit design. After much review during a period of depressed metal prices, it was decided to abandon the proposed underground mining activities and keep the mine dewatered to the 425 m Level. The electrical equipment was removed and the pumps were shut off on 01 April 1998. At present, the mine is flooded above the portal.

Savage concluded that an open pit operation was feasible and initiated environmental permitting, including provisions for a diversion of a portion of the Gays River. The environmental assessment plan was approved August 2000. The operating plan was never initiated, probably due to low metal prices at the time.

ScoZinc, purchased by Acadian Mining (ADA, TSX-V) in 2006 as its wholly-owned subsidiary, continued with Savage's plan and surface-mined the deposit during 2007 and 2008. ScoZinc mined 1.1 million tonnes of surface ore and stripped 9.4 million tonnes of overburden (refer to Table 6-1). Due to a drastic plunge of base metal prices nearly coinciding with the mine's re-opening, ScoZinc placed the mine on care-and-maintenance status near the end of 2008.

In 2008, ScoZinc also drilled 17 diamond drill holes through the Northeast Zone (refer to Section 10.0).

In May 2011, Selwyn purchased ScoZinc with plans to reopen the mine amid high and rising metal prices.

6.2 Ownership History

6.2.1 Gays River Deposit

The Gays River Deposit was discovered in 1973 by the Esso and Cuvier joint venture. Esso initiated mine development in 1978, commissioned the mill in 1979, developed the underground mine, and began mining and milling.

Seabright acquired the Scotia Mine property and mill in 1984. Despite a favorable feasibility study, Seabright did not reactivate the Scotia Mine due to depressed metal prices at the time. Seabright converted the mill for gold processing and processed gold ore from several satellite properties.

The Scotia Mine property was acquired by Westminer Canada Limited (Westminer), a Canadian subsidiary of Western Mining Corp of Australia, in 1988, at which time a review of the potential for mining the deposit was undertaken. Westminer dewatered the mine and continued mining and milling.

In 1997, Savage Resources Canada Limited acquired the Scotia Mine assets from Westminer. Savage concluded that an open pit operation was feasible and initiated environmental permitting, including provisions for a diversion of a portion of the Gays River. Savage was subsequently taken over by Pasminco Resources Canada Company (Pasminco Resources) and the environmental assessment plan was approved by the Nova Scotia Minister of the Environment in August 2000. The operating plan was never initiated.

Regal Mines Limited (Regal Mines) purchased Pasminco Resources in February 2002. Regal Mines was owned 50% by OntZinc Corporation (OntZinc) and 50% by Regal Consolidated Ventures Limited (Regal Consolidated). As part of the sale, Pasminco Canada Holdings Inc. (Pasminco Holdings) retained a 2% net smelter return (NSR) royalty on future production. OntZinc acquired Regal Consolidated's 50% interest in December 2002 to own 100% of Pasminco Resources. Savage Resources Limited was the successor of Pasminco Holdings and held the 2% royalty.

OntZinc later changed its name to HudBay Minerals Inc. (Hudbay) after purchasing, through reverse takeover, Hudson's Bay Mining and Smelting in December 2004. Hudbay owned the Scotia Mine through its wholly-owned subsidiary, ScoZinc Limited.

In 2006, Acadian Gold Corp (Acadian Gold) purchased 100% of ScoZinc and all of its assets (consisting mainly of the Scotia Mine and its infrastructure) from OntZinc for $7 million. Acadian Gold subsequently changed its name to Acadian Mining Limited (Acadian Mining). On 29 May 2007, ScoZinc exercised its option to buy-out the 2% NSR for $1,450,000.

ScoZinc reactivated the mill and continued surface mining the deposit during 2007 and 2008. Depressed metal prices forced ScoZinc to place the mine on care-and-maintenance status.

In May 2011, Selwyn Resources Limited (Selwyn) purchased ScoZinc and all of its assets, including the Scotia Mine and ScoZinc's exploration claims, for $10 million less a deduction relating to increased reclamation bonding requirements that were being determined at the time of the acquisition. In a letter dated 02 May 2011, the Nova Scotia government informed ScoZinc that the increased bond requirement amounted to $1,887,790 (refer to Section 4.3). On 01 June 2011, Selwyn announced the closing of the sale and therefore acquiring 100% of ScoZinc Limited and all of its assets.

6.3 Historical Mineral Resource and Mineral Reserve Estimates

The following resource and reserve estimates are historical in nature, have not been extensively audited by the authors, were not prepared according to NI 43-101 (except where noted), and should not be relied upon.

6.3.1 Gays River Deposit

Numerous resource estimates have been carried out over the past 30 years since the discovery of the Scotia Mine mineralization. These resource estimates have been based on differing underlying parameters, including varying minimum thickness of intercept, differing cut-off grades, utilization of Zn equivalent or independent Pb and Zn minimum grades, etc. Resource figures have ranged throughout the years from an initial 12,000,000 t at 7% Zn-equivalent (drill-indicated) in 1974 (Patterson, 1993) to the 1985 figure of 980,000 t at 5.35% Pb and 9.42% Zn (mineable) at a 7% Zn-equivalent cutoff (Hale and Adams, 1985).

Westminer (Nesbitt Thompson, 1991; WMC, 1995) reported resources that were outlined by over 1,300 underground and surface holes in addition to the information derived from the underground workings. The calculations were based on a minimum true thickness of 2 m with a cutoff of 7% Zn-equivalent. The total geologic reserves were quoted as 2,400,000 t averaging 6.3% Pb and 8.7% Zn (Table 6-2). A mineable reserve was also quoted as 1,370,000 t, averaging 5.3% Pb and 9.8% Zn.

In 1992, Campbell, Thomas, and Hudgins reported that there was potential for mining an additional 800,000 t of lower grade mineralization via open pit methods. The authors went on to say "there is excellent potential to expand the underground reserves, particularly in the eastern section of the mine. Underground development in the western and central zones resulted in significant expansion of the reserves as ore zone continuity has generally been better than had been originally interpreted from the drill information."

In Claude Poulin's 01 July 1998 memo titled "Scotia Mine, Mineral Resource Status," he reported the deposit's resources. Higher grade (i.e., greater than 7% Zn-equivalent [% Zn + 0.5 × % Pb]) and lower grade (greater than 2% but less than 7%) zones were outlined by Savage's geologists. The higher-grade zone consists of massive sulfide and lies at the contact between the dolomite and the trench or evaporite units. The lower grade zone consists of disseminated Zn and Pb within the dolomite. These outlines were transferred to a block model by Tim Carew, manager of Gemcom Services in Reno, Nevada. Inverse-distance squared weighting was used to calculate block grades. Top-cut values of 15% Zn and 10% Pb were used. No dilution or mining recovery factors were applied to the calculations. Undiluted resources are reported in Table 6-2.

The reader should note that the resources were unclassified. They were not separated into Measured, Indicated, and Inferred categories "due to the lack of geostatistical information" [Poulin, 1998 (1)]. Those resources were not entirely independent and did not follow NI 43-101 guidelines, as the report predated that standard.

Reserves were estimated through a pit optimization process carried out on the central portion of the deposit. These were reported in Claude Poulin's 01 July 1998 memo titled "Scotia Mine, Mining Reserve Status." Zn and Pb prices were US$ 0.55 and US$ 0.36 per pound, respectively. The optimized pit, which considered diverting Gays River by moving it toward the highway, was sent to Mine Design Associates (MDA) for practical pit design. Savage supplied the economic and geotechnical parameters to MDA. Dilution and recovery factors of 20% and 90%, respectively, were used.

Reserves included resources that lie northeast of the highway. These would be accessed using underground methods. For this material, dilution, and recovery factors of 25% and 90%, respectively, were used. The estimated Reserves are reported in Table 6-2. Those Reserves were not entirely independent and did not follow NI 43-101 guidelines, as the report predated the standard.

Table 6-2: Historical Resource and Reserve Estimates

Estimator	Category	Tonnes	Zinc Grade	Lead Grade
Westminer (1991)	"Geologic Reserve" (Undifferentiated)	2,400,000	8.7%	6.3%
	Reserve (Underground)	1,370,000	9.8%	5.3%
Savage (1998)	Resource (Undifferentiated):			
	Higher Grade	1,700,000	11.1%1	4.7%1
	Lower Grade	3,400,000	2.6%1	1.3%1
	Total	5,100,000	5.5%1	2.4%1
	Reserve (Undifferentiated):			
	Northeast (Underground)	360,000	8.6%	4.3%
	Central (Open Pit)	1,900,0001	4.1%1	1.6%1
	Total	2,260,000	4.8%	2.0%

It should be noted that the above referenced historical Resources and Reserves estimates were not carried out in accordance with the Canadian Institute of Mining and Metallurgy and Petroleum CIM standards on Mineral resources and Reserve Definitions ("CIM Standards") and therefore do not conform to Sections 1.3 and 1.4 of NI 43-101.

It was discovered during the current Resource estimation process that an error was made when calculating resource and reserve grades during the 1998 estimate. When estimating block grades in the high-grade zone, lower grade (less than 7% Zn-equivalent) assays were filtered out because they were thought to belong to a separate domain. Likewise. in the lower grade disseminated zone, higher grade (greater than 7% Zn-equivalent) was filtered out. This incorrectly increased the grade of the high-grade zone, which increased the overall resource and reserve grade by approximately 1% Zn-equivalent. The error had less of an effect on the lower grade zone. The error was corrected during the current Resource estimate.

In 2006, MineTech International Limited (MineTech) carried out a NI 43-101-compliant resource and reserve estimate. MineTech's results are reported in Table 6-3.

Table 6-3: Previous Mineral Resource and Reserve Estimate (Roy et al., 2006)

Mineral Resources					
Category	Volume (m^3)	SG	Tonnes	Zinc Grade	Lead Grade
Measured (surface)	680,000	2.78	1,880,000	3.80%	1.60%
Indicated					
Surface	810,000	2.77	2,250,000	3.20%	1.40%
Underground	381,000	2.9	1,110,000	6.60%	3.70%
Subtotal	1,190,000	2.82	3,360,000	4.30%	2.20%
Measured and Indicated (Surface and Underground)	1,870,000	2.8	5,240,000	4.10%	2.00%
Inferred	652,000	2.76	1,800,000	3.10%	1.10%
Mineral Reserves					
Category	Volume (m^3)	SG	Tonnes	Zinc Grade	Lead Grade
Proven Reserves (Surface)	630,000	2.78	1,750,000	3.20%	1.30%
Probable Reserves					
Surface	610,000	2.76	1,690,000	2.50%	1.00%
Underground	395,000	2.9	1,150,000	5.70%	3.20%
Subtotal	1,005,000	2.83	2,840,000	3.80%	1.90%
Total Proven and Probable Reserves (Surface and Underground)	1,635,000	2.81	4,590,000	3.60%	1.70%

6.4 Mine Status, Activity, and Changes—2012 To Present

Since mid-2013, the ScoZinc Mine has been in a state of Care and Maintenance, in an effort to conserve resources in response to the continuing challenges facing the mineral exploration and mining industry. The limited activity carried out at the mine since that time has been focused on facilitating a return to full production status. To that end, the ScoZinc Care and Maintenance program has primarily intended to maintain the value of the property, in terms of protecting the integrity of physical structures and land, and ensuring that the applicable permits, approval, licenses remain in good standing through environmental monitoring work and administrative oversight. Other activities intended to add value, such as the production of an updated mineral resource estimates and PEA, and upgrades to the mill hardware and equipment, have been carried out as well.

ScoZinc continues to own property on and around the mine area. While these properties are unchanged since 2012, a full list of ScoZinc properties is provided in Section 4.3. ScoZinc continues to operate as a company in good standing in the Province of Nova Scotia.

6.4.1 Security

The ScoZinc staff presence at the mine on a day-to-day basis consist largely of security personnel, who carry out regular patrols and maintain watch over the property 24 hours per day, 7 days a week.

6.4.2 Environmental Programs

Regular environmental monitoring work, in accordance with the terms of ScoZinc's Industrial Approvals and the Federal Metal Mining Effluent Regulations, has been carried out since 2012. A full summary, outlining the details of this ongoing work is provided in Section 20.0, Environmental Studies, Permitting, and Social or Community Impact.

6.4.3 Technical Work

Technical work has included the development and publication of an updated mineral resource estimate 43-101, which was filed on SEDAR in October 2012 by Selwyn Resources Ltd., the parent company of ScoZinc Limited. (Note: Selwyn Resources Limited has changed its name to ScoZinc Mining Ltd., effective 01October 2015.) This document, written by MineTech International Limited in conjunction with ScoZinc Limited, updated the mineral resource estimate for both the Gays River and Getty Deposits.

Technical work has also included the development of a PEA 43-101, which was filed on SEDAR by Selwyn Resources Ltd. in December 2012. The PEA was updated and filed again in June 2013.

Further technical work has largely consisted of the continued development of the Preliminary Reclamation Plan. Significant work was completed on this in mid-2013, which has been synthesized into a final Preliminary Reclamation Plan in Q4 -2016.

6.4.4 Mill Upgrades

Upgrades to the mill facility were carried out in 2012 and 2013, including the purchase of a screening plant, the refurbishment of the floatation circuit system, and the integration of numerous pumps. Regular maintenance of the mill facility is carried out as well, ensuring that the physical structure is kept in working condition.

6.4.5 Business Development

Another primary focus of ScoZinc since 2012 has been on business development, in an effort to return the mine to production. To that end, ScoZinc has engaged with numerous potential business partners and suitors over the past several years, much of which has involved physically showing property, as well as presenting and explaining the details of the deposit and proposed operation.

6.4.6 Regulator Liaison

Since the operation went into non-operation in 2012 ScoZinc has maintained an open and cooperative relationship with all regulatory agencies with a role in the operation. Nova Scotia Department of Natural Resources (NSDNR) and Nova Scotia Environment (NSE) have conducted routine inspections and site visits with any minor issues identified being addressed to their satisfaction.

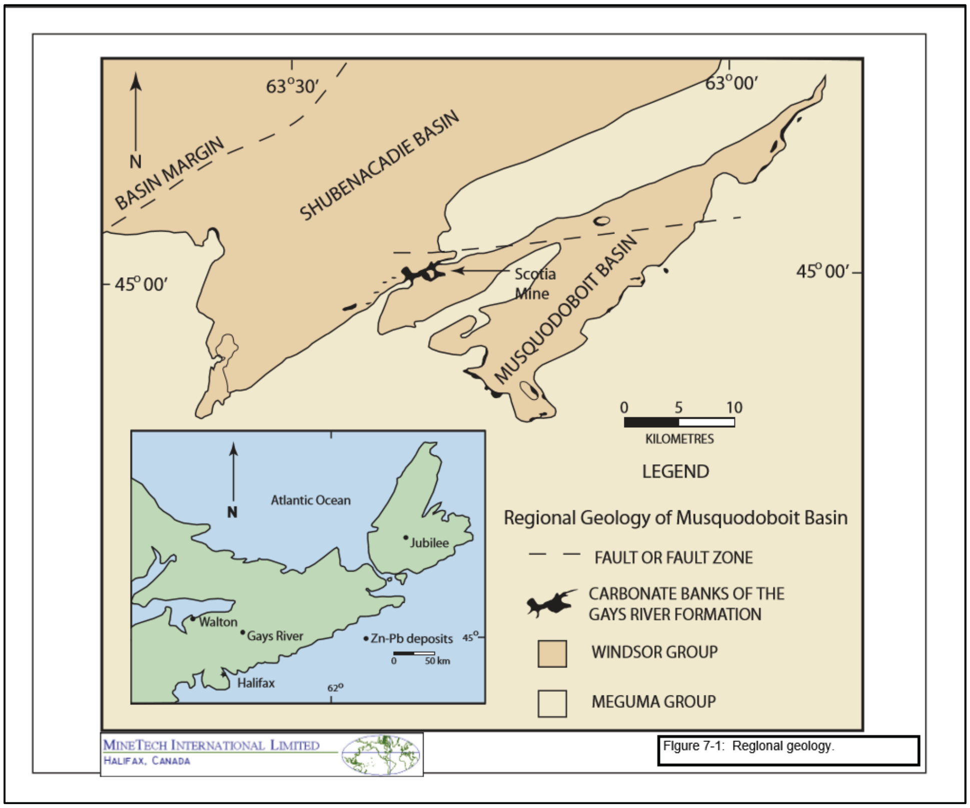
7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional and Local Geology

An excellent summary of the regional and deposit geological settings of the Gays River area was supplied by Patterson (1993). There is also a recent “special issue devoted to Zn / Pb mineralization and basinal brine movement, lower Windsor Group (Viséan), Nova Scotia Canada” released as Volume 93 by Economic Geology in 1998. The bulk of the descriptions below are taken from those publications.

The Gays River and Getty Deposits occur along the southern margin of the large (more than 250,000 km^2) and deep (more than 12 km) late Palaeozoic Fundy (Magdalen) Basin, bordered on the northwest by the New Brunswick platform, and on the south by the Meguma platform (see Figure 7-1). During the late Palaeozoic, the Fundy Basin was divided or segregated through a complex series of grabens into deep linear successor basins or sub-basins, which are now interpreted (Fralic and Schenk, 1981) as small pull-apart basins. Subsequent basement subsidence, fragmentation and block faulting produced the irregular pre-Carboniferous topography that was partly filled-in by early Carboniferous clastics, and later flooded by middle Carboniferous seas. Carboniferous sediments consisting of terrestrial conglomerates, sandstones, siltstones, and marine limestones and evaporites, were deposited in this Fundy Basin, which probably remained active during and after the Carboniferous, and may have had a major impact in the ore-forming process. These sub-basins contained thick accumulations of terrestrial and shallow marine sediments, and therefore could provide substantial volumes of basinal fluids (Ravenhurst, 1987).

Figure 7-1: Regional Geology



In their 2011 report, Cullen et al. give further detail about the Carboniferous strata:

> *"The Getty Deposit is hosted by lower Mississippian age dolostone of the Windsor Group's Gays River Formation. Well defined carbonate banks characterize this formation and in most instances are associated with well-defined paleo-basement high features. On depositional basin scale, Gays River Formation bank carbonates and laminated limestone of the laterally equivalent Macumber Formation mark the onset of marine depositional conditions after a prolonged period of predominantly terrigenous clastic sedimentation represented by Horton Group siliciclastic rocks.*
>
> *"Carboniferous strata in Central Nova Scotia occur within the Shubenacadie and Musquodoboit sub-basins of the larger Maritimes basin and were described by Giles and Boehner (1982). Geometry of both sub-basins was significantly influenced by strong northeast trending structural grain in basement sequences of the Cambro-Ordovician Meguma Group. Deformation was heterogeneously distributed across the sub-basins and at present is now represented by northeast trending normal and thrust faults which are locally associated with open to moderately folded structural domains. Deformation features are essentially absent near the southern margins of the basins but become more prevalent and*

pervasive toward the northern limits, where effects of the regionally significant Cobequid-Chedabucto fault system are represented. Minor faults or fracture zones may be present at Getty but no structural complexity is evident in either the surface morphology or drill logs." (Cullen et al..,., 2011, Section 6)

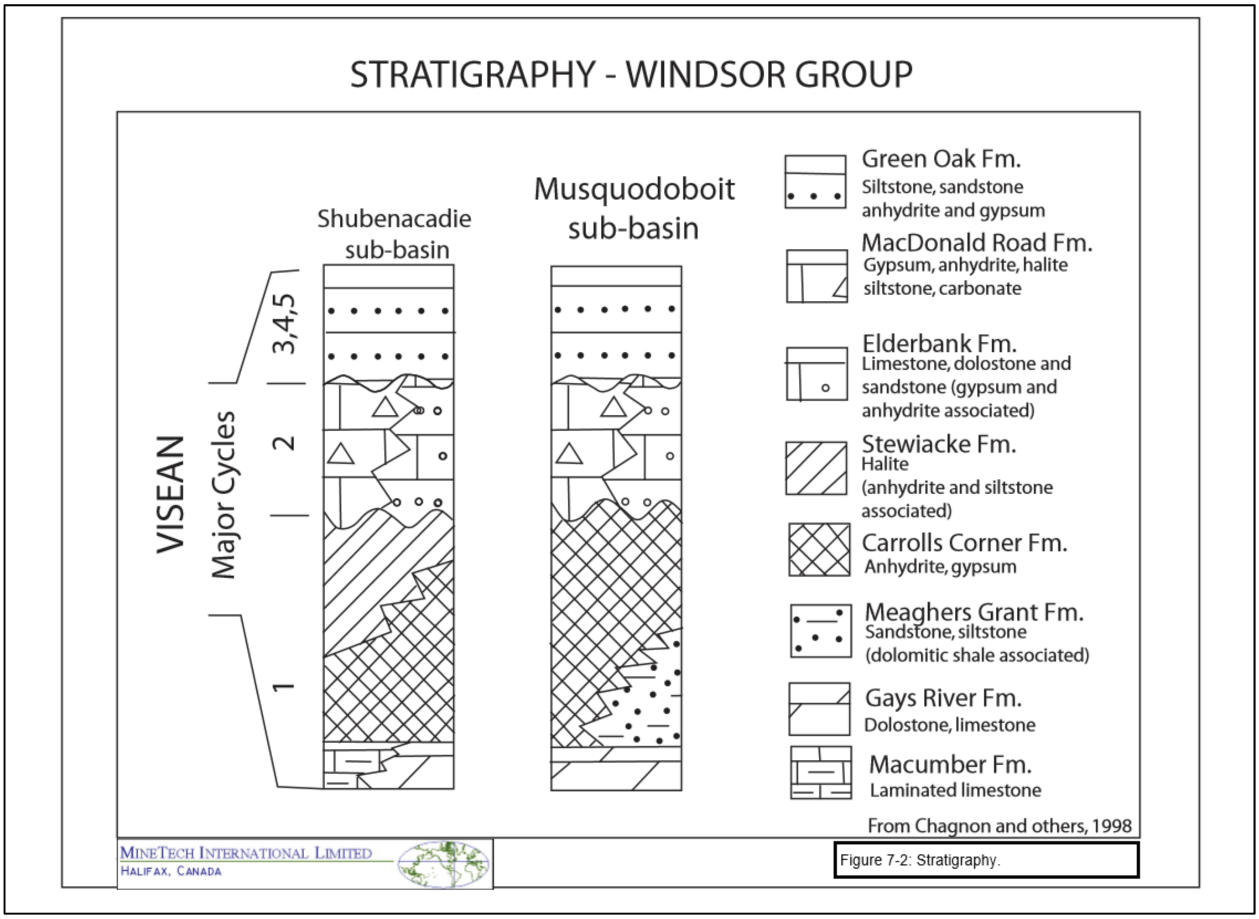
The Gays River area is underlain by the Cambro-Ordovician metasediments of the Meguma Group which form the pre-Carboniferous basement upon which the Gays River carbonate host rock was deposited. The Meguma rocks were tightly folded during the Acadian Orogeny into long northeast-southwest anticlines and synclines which have been faulted and jointed. Erosion of this basement into irregular knobs and ridges was controlled by these structures prior to the deposition of overlying sediments (the Gays River carbonate). Unconformably overlying the Meguma Group are clastic sedimentary rocks of the Horton Group and marine sedimentary rocks of the Windsor Group. The Windsor Group overstep the Horton near the basin margins and rest directly on Meguma basement. It is these Windsor Group carbonates which have been the host for the carbonate-hosted base metal sulphide and associated sulphate deposits in Nova Scotia.

Over 100 base metal occurrences, including a few deposits, are hosted by Lower Windsor Group marine carbonate rocks in Nova Scotia. About half of these occur within the Kennetcook, Shubenacadie, Musquodoboit, and River Denys sub-basins. In addition to the Gays River and Getty Deposits, the most significant examples include the Walton deposit and the Jubilee deposit. Walton has two types of mineralization: concordant sheets of barite contain lenses of Pb-rich and copper-rich mineralization. Between 1941 and 1978, 4.5 million tonnes containing over 90% $BaSO_4$, and 0.4 million tonnes containing 0.52% Cu, 4.28% Pb, 1.29% Zn, and 350 g/t Ag were produced (Sangster, Savard and Kontak, 1998). At the Jubilee deposit on Cape Breton, sulphides cement fault-related breccias and replace adjacent limestone; there are reported, unclassified resources (e.g., Fallara and Savard, 1998) of 0.9 million tonnes containing 5.3% Zn and 1.4% Pb.

7.2 Property Geology

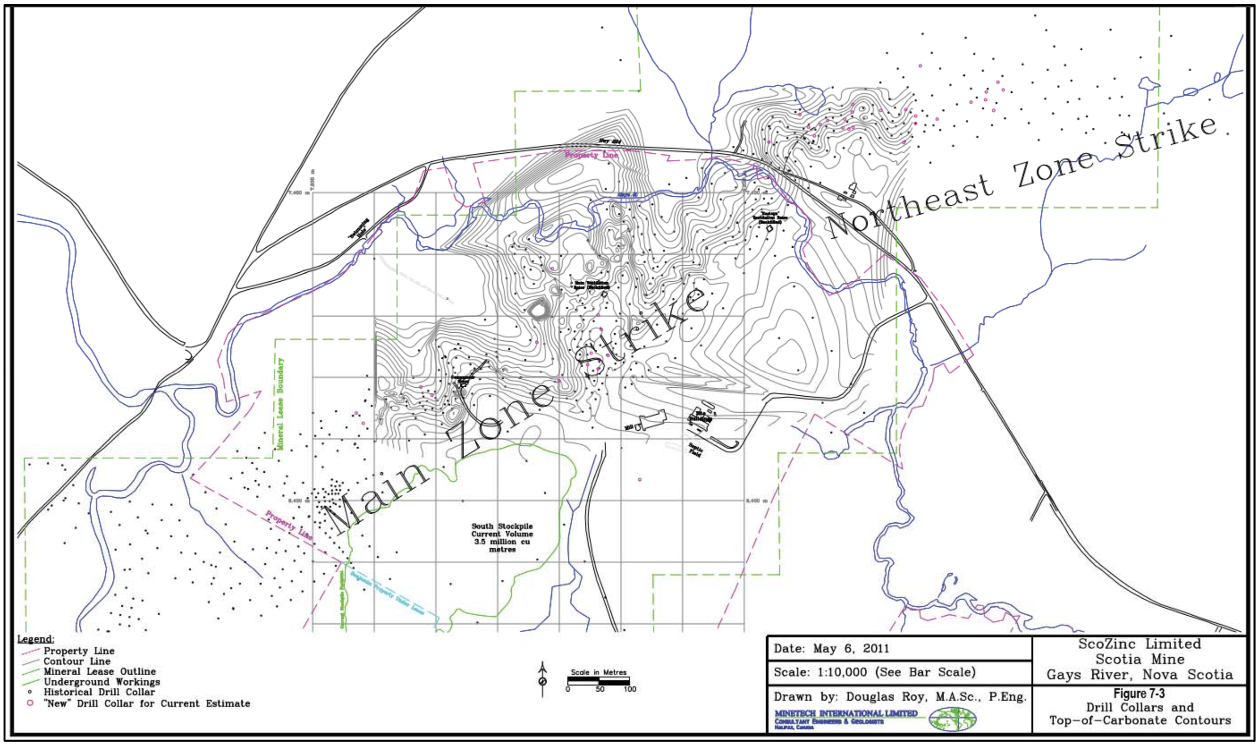
The Gays River Formation and its lateral equivalent, the Macumber Formation, form the basal carbonate units of the Windsor Group. There is an angular unconformity between the marine sediments (Gays River Formation and Macumber Formation) and the underlying basement rocks. The underlying 380-400 million-year-old basement rocks consist of greenschist facies meta-turbidites of the Meguma Group that form a northeast-trending, paleotopographic high which separates the Shubenacadie and Musquodoboit basins, and over which the Gays River carbonate bank developed (Kontak, 1998; Savard & Chi, 1998). The generalized regional Windsor Group stratigraphy is shown in Figure 7-2.

Figure 7-2: Stratigraphy



The basement is overlain by a laterally extensive, but discontinuous, talus breccia composed of centimeter- to meter-size, rounded to sub-rounded fragments of Meguma Group lithologies cemented by dolostone. Overlying the basal breccia, or directly in contact with the basement rocks, is a carbonate build-up composed of various bank and interbank facies: algal, coral and bryozoan bafflestones, skeletal packstones, and wackestones. The carbonate bank can be traced basinward into a laterally extensive, thinly laminated, 3 m to 18 m thick argillaceous, bituminous dolostone or limestone unit referred to as the Macumber Formation. Contours for the top of the carbonate are showing in Figure 7-3.

Figure 7-3: Top Carbonate Contours



Overlying the carbonate rocks are evaporites (gypsum, anhydrite, halite, and minor potash) with minor interbeds of dolostone and mudstone, all of which constitute the Carroll's Corner Formation. Nearby (5 km to the southwest), the gypsum is being mined at the National Gypsum Quarry.

In the deposit area, the contact between the evaporites of the Carroll's Corner Formation and the carbonates of the Gays River Formation was deeply incised by a palaeochannel during a period of uplift and erosion during the Cretaceous period. It was filled-in by sedimentary debris (boulders, sands, silts, clay, and gypsum fragments) to which a Cretaceous age has been assigned. This dense, over-compacted debris has been termed trench material; it occurs adjacent to the massive sulphide mineralization. Near the contacts, highly permeable, open channel-type structures have caused locally high rates of water flow that have been an impediment to underground mining.

Both the bedrock and trench sediments are overlain by 20-40 m of glacial till, which is locally cut by glacial-fluvial sands and gravels. Three geological cross sections are included as Figure 7-4, Figure 7-5, and Figure 7-6. represents the prototypical cross-sectional geology for the deposit.

Figure 7-4: Section A---A′

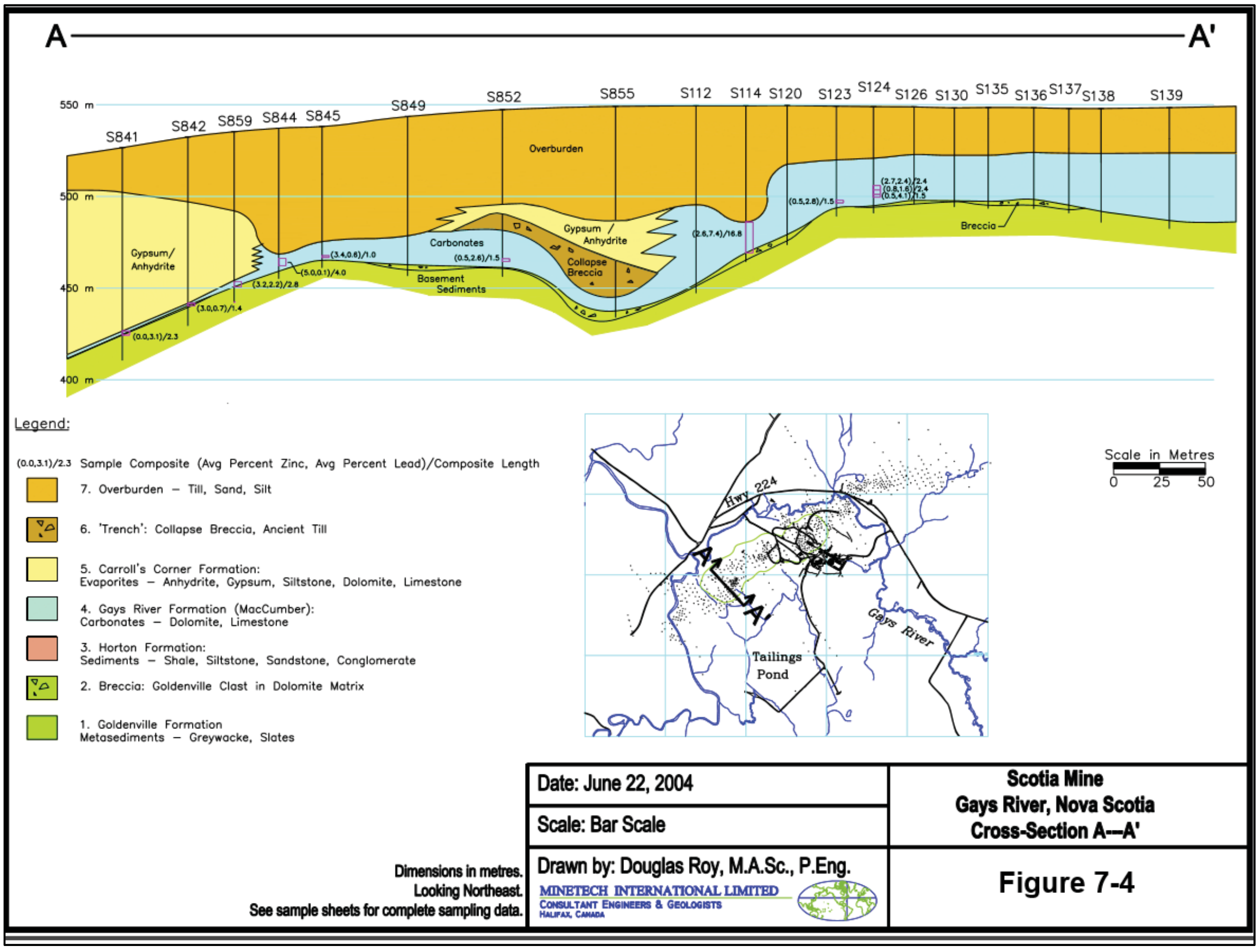


Figure 7-5: Section B---B′

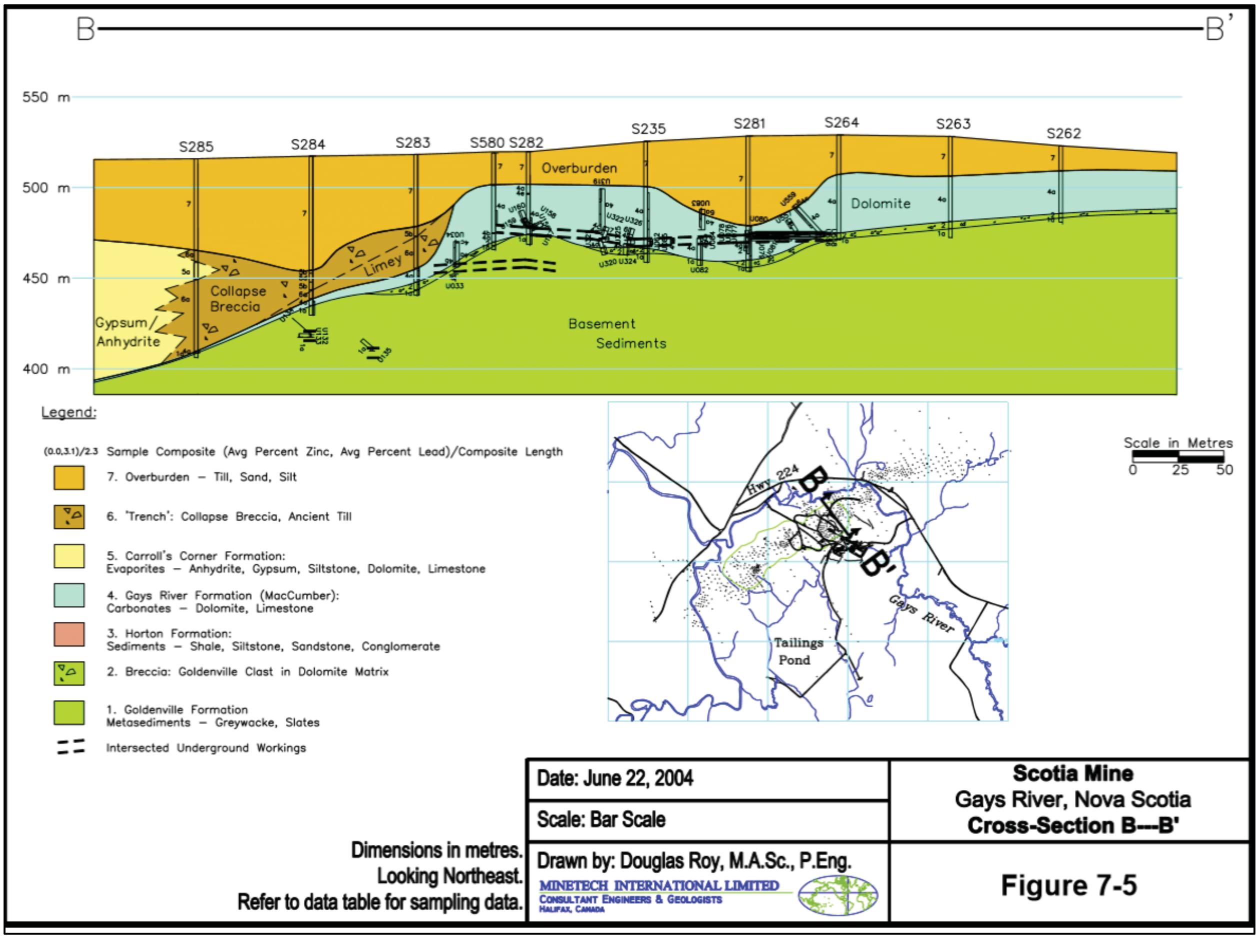
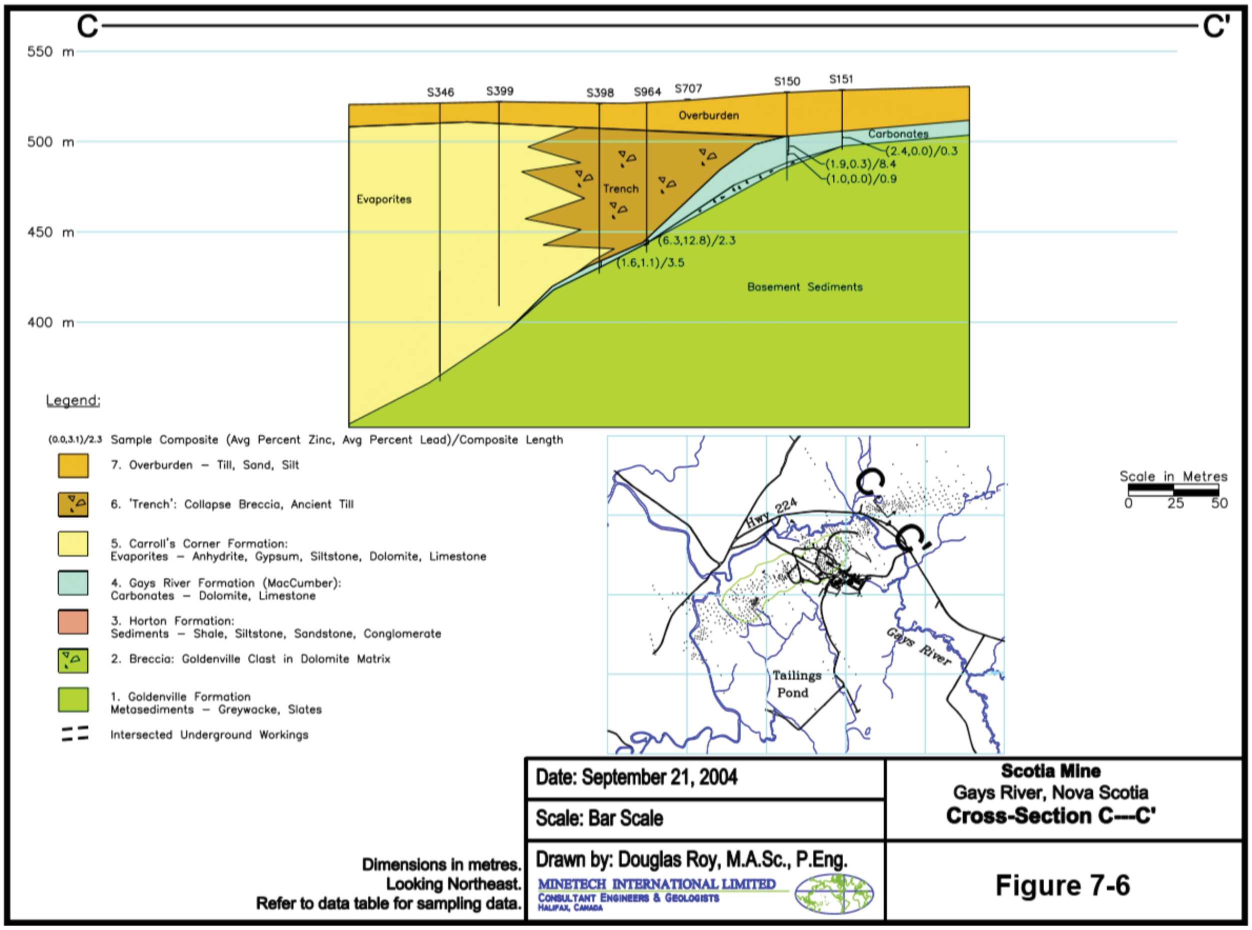


Figure 7-6: Section C---C′



Cullen et al. (2011) describe the Getty Deposit in Section 7.1 & 7.2 (‘Stratigraphy’ and ‘Deposit Type’, respectively) of their report:

> *“Geology in the Getty Deposit area has been interpreted from compiled results of Giles and Boehner (1982) plus results of various mapping and diamond drilling campaigns carried out in the area. The actual deposit does not outcrop, but was delineated by Getty through drilling (e.g., Bryant, 1975, Comeau, 1973, 1974; Palmer and Weir, 1988a, b).*
>
> *“As represented in [Figure 7-7], the Getty Deposit is hosted by a northwest trending Gays River Formation carbonate bank complex that occurs as a direct extension to the larger, northeast trending carbonate bank that hosts Scotia Mine's zinc lead resources and reserves. Both banks developed along paleo-basement highs comprised of Cambro-Ordovician age Goldenville Formation quartzite and greywacke. At Getty host dolostone ranges in true thickness from less than a meter to a maximum of about 45 meters.*

"The carbonate host sequence occurs above a thin sedimentary breccia or conglomerate unit comprised predominantly of Goldenville Formation debris with a small carbonate matrix component resting unconformably on Goldenville Formation basement. Carrolls Corner Formation evaporites lie stratigraphically above the Gays River Formation and are comprised locally of gypsum and anhydrite with minor amounts of interbedded dolomitic limestone and siltstone. With possible exception of local clay and sand accumulations of Cretaceous age, Carrolls Corner Formation rocks are the youngest sequences of the local bedrock section. Figure 7-8 presents a stratigraphic column for the deposit area.

"Historical and recent drilling on the Getty property has shown that evaporite cover at the Gays River Formation contact was in many instances preferentially removed by erosion and karst-related solution processes during Cretaceous time, leaving a trough or trench parallel with the carbonate contact in many areas. Stratified Cretaceous fill sedimentary material followed by Quaternary material of glacio-fluvial origin infilled this trough, and is termed "Trench" material on the adjacent Scotia Mine property. Similar material exists in some areas adjacent to the Getty Deposit but in many instances is difficult to distinguish from less consolidated overburden material that is of glacial origin.

"The Getty Deposit carbonate bank forms a northwest extension to the adjacent Gays River bank that hosts Scotia Mine zinc-lead resources and reserves. While broadly similar, carbonate bank slopes at Getty are generally gentler than those seen at Gays River. Figure 7-9 depicts a typical bank cross section illustrating occurrence of thickest carbonate on the bank top, with progressive thinning down dip on the paleo-topographic high. Variations existed locally in basement paleo-slope angles and appear to have directly influenced corresponding carbonate bank morphology. Areas with steep basement slopes tend to show rapid thinning of carbonate away from the thicker bank tops, with correspondingly steep contact surfaces with overlying evaporites. Gentle slope areas show greater lateral and down-dip continuation of thicker carbonate and corresponding lower average contact dips with the overlying evaporite. Based on the drilling carried out to date at Getty, the maximum carbonate thickness encountered along the basement high trend is 45.48 meters in drill hole GGR-221.

"Gays River Formation carbonate banks include intricately intercalated algal, peloidal and coraline lithofacies, with abundance of bindstone, bafflestone, packstone and micrite. These facies show transition downdip to thin (typically <5 meters), variably laminated algal/silty carbonates that are lateral equivalents to laminated carbonates of the Macumber Formation. The latter occurs basinward of the underlying Horton Group's stratigraphic pinchout and is not present in the deposit area."

Figure 7-7: Getty Property Geology Map and Deposit Outline (Cullen et al., 2011)

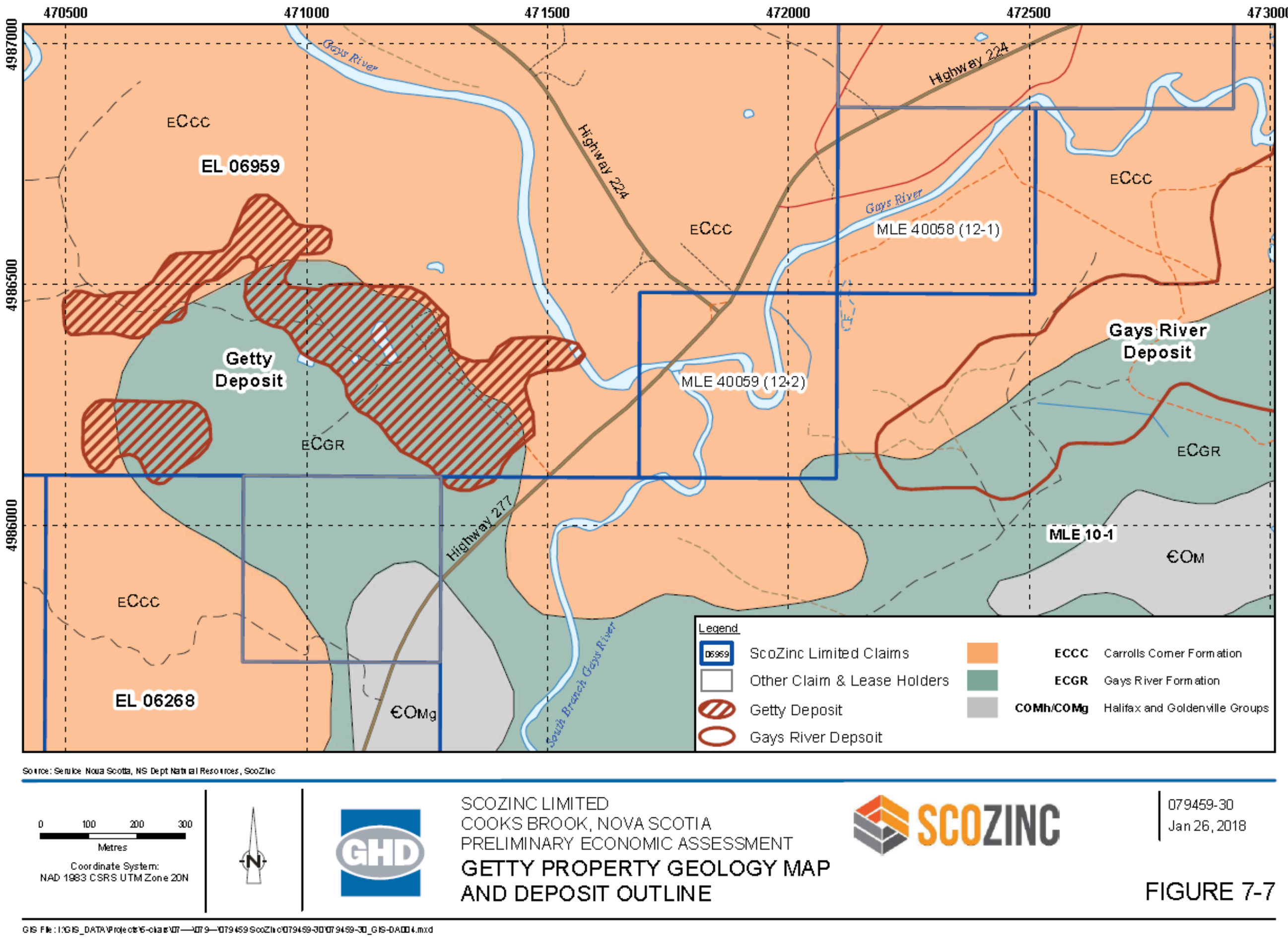


Figure 7-8: Stratigraphic Column for Getty Deposit Area (Cullen et al., 2011)

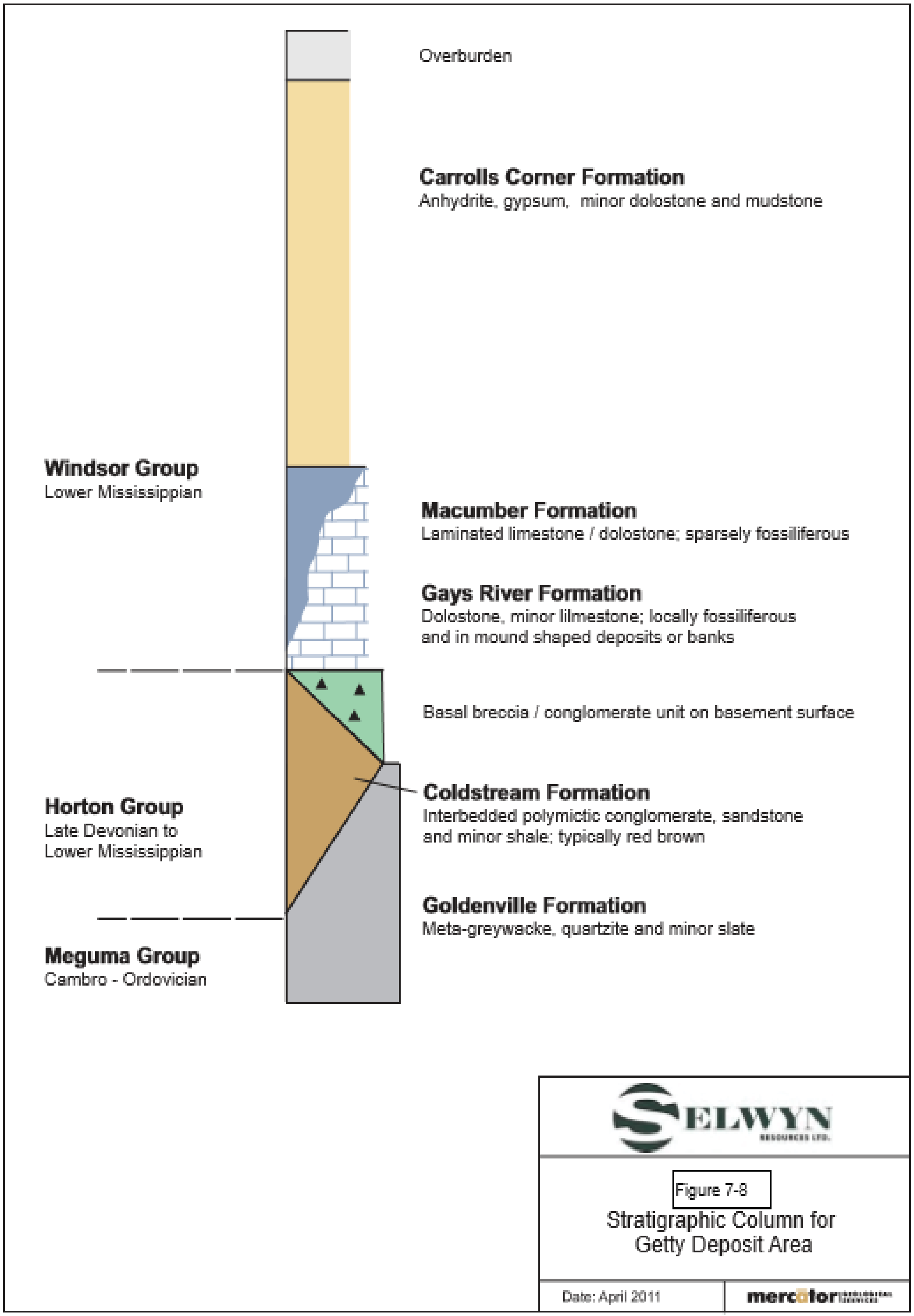
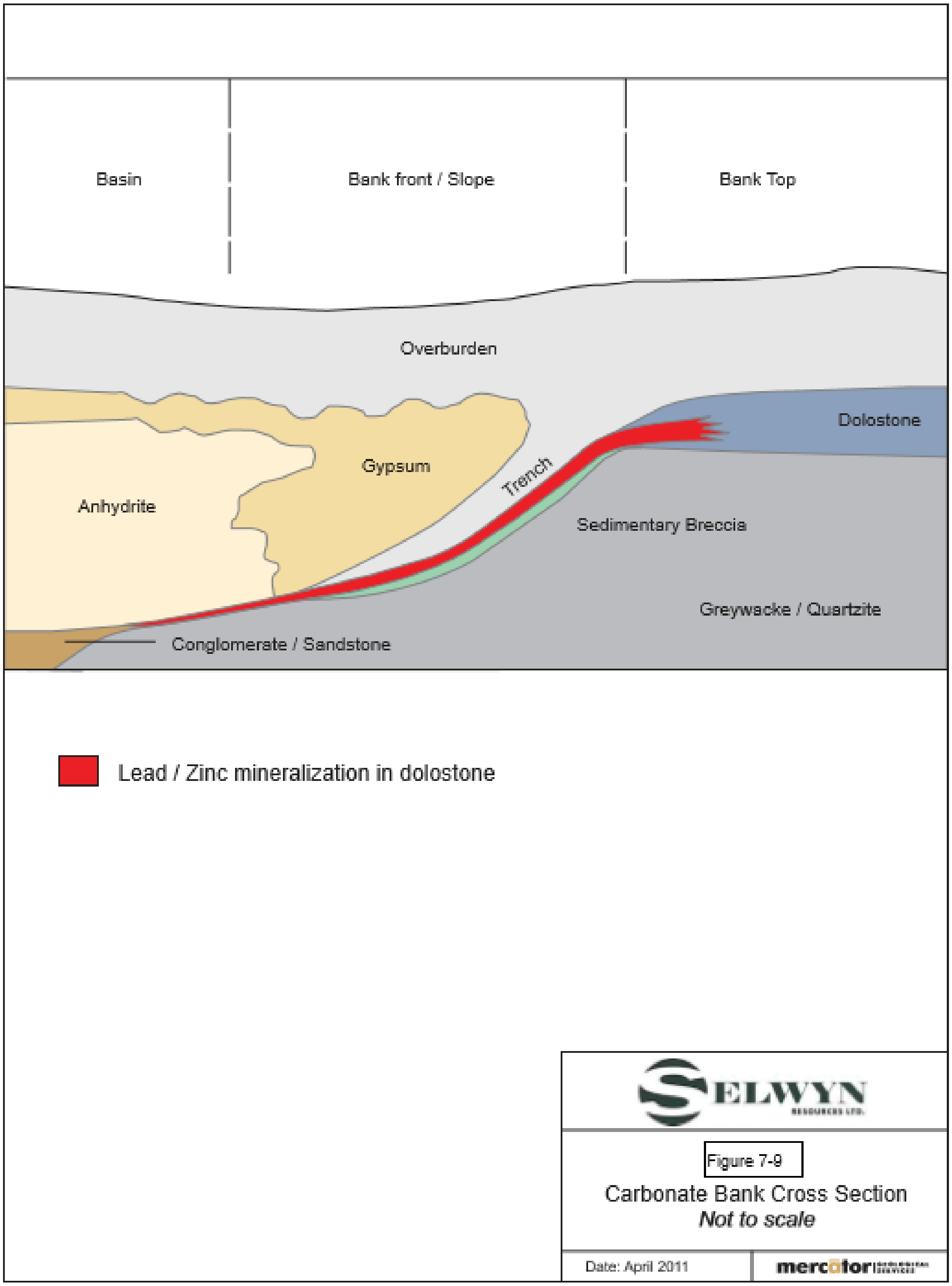


Figure 7-9: Carbonate Bank Cross-Section (Cullen et al., 2011)



7.3 Mineralization

7.3.1 Gays River

Nesbitt Thomson Inc. (1991) describe the high-grade mineralization as consisting of a massive sulphide zone in contact with the evaporite or trench, ranging in thickness from 0.1 m to 5.0 m and locally containing up to 78% Pb and 57% Zn. On the footwall of the massive sulphide, there is a zone of disseminated material (>7% Zn equivalent) which, in places, is up to 12 m in thickness. Locally disseminated mineralization (>2% Zn equivalent) extends up to 20 m into the footwall.

The Gays River Deposit is essentially controlled by a sinuous paleo coastline. The main part of the deposit is shallow (generally <150 m deep), has a dip length of approximately 100 m and a strike length following the paleo-coastline over a straight line distance of 2 km (Nesbitt Thomson Inc., 1991).

The mineralization at the Gays River Deposit consists of massive and/or disseminated ore hosted predominantly by the carbonate rocks, with extensions down into the basal breccia unit. The massive mineralization consists of fine-grained (<10-20 mm), Fe-poor, beige-colored sphalerite and medium to coarse-grained, Ag-poor galena (<10-20 ppm Ag in galena concentrates) (Kontak, 1998; Savard and Kontak, 1998), is restricted to the carbonate-evaporite contact and is 1 m to 3 m in true thickness. Disseminated mineralization, consisting of yellow to orange, millimeter-size euhedral sphalerite and millimeter- to centimeter-size euhedral galena, fills in primary porosity in the dolomitized carbonates and walls of primary cavities (Kontak, 1998).

Sphalerite and galena constitute about 99.5% of metallic minerals. Other sulphide minerals are marcasite, pyrite, and chalcopyrite, while gangue minerals include calcite, dolomite, fluorite, barite, and selenite (Patterson, 1993).

8.0 DEPOSIT TYPES

The Gays River Deposit mineralization has long been considered a Mississippi Valley-type (MVT) Zn / Pb deposit. Characteristics of sedimentary formations that host MVT Zn / Pb mineralization include shallow-water, shelf-type carbonate rocks with reefs around the peripheries of intracratonic basins, karst structures, limestone-dolomite interfaces, and proximity to a major hydrocarbon-bearing basin. The archetypical MVTs occur in the United States in several famous districts surrounding the Michigan-Illinois Basin, which also has significant hydrocarbon production. Each of the districts is enormous, with resource potential of 75 million tonnes to 750 million tonnes and individual deposits in the order of 1 million tonnes to 100 million tonnes.

Other MVTs have been mined in the past in Canada (e.g., Pine Point in the Northwest Territories, Nanisivik mine in Nunavut, and Newfoundland Zinc) and in Ireland.

MVTs are thought to have formed when hot, basin-derived, oil field-type brines formed at depths of more than 2 km, migrated towards lower pressure areas around the basin periphery. Mineralization precipitated from the brines when they encountered porous areas like reefs, karst breccias or sedimentary traps.

Sangster and others (1998) draw on their own and other's evidence to conclude that all Windsor Group Zn / Pb deposits are epigenetic relative to their enclosing strata, exhibiting both open-space filling and host-rock replacement. At the Gays River Deposit, textures (including fossils) have been preserved; representing volume-for-volume replacement of original limestones by dolomite, and the sulphides are, in turn, replacements and porosity fillings within the previously altered host rocks. Kontak (2002) feels that petroleum in fluid inclusions in the Gays River Deposit mineralization suggest a role of hydrocarbons in the mineralizing process, like many MVTs, but Sangster and others (1998) point to basement rocks underlying the Palaeozoic sedimentary rocks as the source of the mineralizing fluids.

The temperatures of formation of the Gays River Deposit (and others in Nova Scotia) are higher than most North American MVTs, and compare more favourably with the clearly epigenetic MVTs of the Central Ireland Basin (Sangster and others, 1998). The Irish deposits also occur in Upper Paleozoic (Carboniferous) carbonate rocks, predominantly in shallow-water carbonates and a mudbank limestone (reef). The Irish deposits are also preferentially associated with east-northeast-trending faults, which are thought to have acted as conduits for mineralizing hydrothermal fluids; basement lineaments may also have controlled deposition. As with the Gays River Deposit, sphalerite and galena are the main sulphides; barite is also usually present (Exploration and Mining Division Ireland, 2004). Seven economic deposits have been mined or are currently in production in Ireland. The largest of these, the world-class

Navan deposit, had total production and proven plus probable reserves of 82.1 million tonnes containing about 10.6% Zn plus Pb; its annual production is 2.5 million tonnes of ore. Other producers and former producers had resources between about 8 million tonnes and 18 million tonnes and grades of 9%-25% Zn plus Pb (Exploration and Mining Division Ireland, 2004).

It is worth noting that two major carbonate-hosted Zn / Pb deposits discovered in Ireland after 1986, occur down-dip from areas where considerable exploration, including diamond drilling, had been carried out over the prior 20 years (Patterson, 1993). Similarly, the MVT deposits of the Viburnum trend in the U.S.A. were discovered at depths of 300 m by understanding the regional geology of the hosts rocks of the Old Lead Belt about 80 km away.

Cullen et al. (2011) describe the Getty Deposit in Section 7.2 of their report, quoted below in part:

> *"The adjacent Scotia Mine deposit (Gays River Deposit) has been the subject of extensive academic and government research and reporting since its discovery in 1971. Much of this work was summarized by Roy et al. (2006) and the deposit is considered an example of the Mississippi Valley Type (MVT) class of carbonate hosted, strata bound, base metal deposits. Prominent examples of the paleo-basement high deposit setting occur along the Viburnum Trend of Southeast Missouri, but are characterized in that area by dominance of lead mineralization over that of zinc (Sangster et. al., 1998; Akande and Zentilli, 1983; MacEachern and Hannon, 1974).*
>
> *"Localization of base metals within the Getty bank complex is believed to have resulted from interaction between metal-bearing basinal fluids, potentially sourced in the Horton Group stratigraphic section or in basement sequences, and chemical reductants, possibly including hydrocarbon, that were present at sites of deposition within the bank. Kontak (1998, 2000) reported on fluid inclusion and other studies of ore from the adjacent Scotia Mine property and concluded that saline brines in the 100° C to ≤ 250° C temperature range were involved in the main mineralizing process and that these temperatures are higher than those typically seen in MVT districts. Héroux et al (1994) studied organic maturation and clay mineral crystallinity characteristics of Gays River Formation rocks of the Musquodoboit and Shubenacadie basins and identified a corridor of higher interpreted heat flow that occurs in part over the Gays River and Getty Deposit areas and is consistent with the higher fluid temperatures previously noted. It is clear that zinc and lead mineralization were superimposed on lithified and dolomitized host rocks (Akande and Zentilli, 1985; Kontak, 1998)." (Cullen et al., 2011)*

9.0 EXPLORATION

The Gays River and were explored more-or-less contemporaneously. Major drilling campaigns on both deposits first started in the mid-1970s. Esso Minerals was primarily involved with the Gays River Deposit, while Getty Northeast Mines Limited was primarily involved with the Getty Deposit. During the 1980s, Seabright and Westminer carried out some drilling on the Gays River Deposit, and during the late-2000s, ScoZinc chiefly drilled the Getty Deposit.

9.1 Gays River Deposit

Zn / Pb mineralization at Gays River was first mentioned in records dating back to 1824. Knowledge of the occurrence may even go back to the early 1700s, when French soldiers reportedly used the Pb for making ammunition (MacEachern and Hannon, 1974). Other early references to Gays River Pb were made in 1868 by J. W. Dawson in “Acadian Geology”, and by H. Howe in “Mineralogy of Nova Scotia”.

The earliest recorded prospecting may have been trenching along the outcrops in 1873-1874. Additional trenching and pit sinking was carried out in 1928. Assessment records do not indicate any resumption of interest in the area until 1951. From the first reports of mineralization in the area in the early 1800s, exploration activity up to 1950 had yielded best values of 3% Pb (Patterson, 1993).

9.1.1 Timeline: 1951

Maritime Barytes Limited acquired the property at Gays River and carried out a surface exploration program involving some trenching and sampling. Gays River Lead Mines Limited subsequently became involved in the evaluation of the property and commenced a drill program to delineate the occurrences of Zn and Pb. A total of 67 delineation drill holes were completed by mid-1952 and an additional 7 holes were completed for exploration in the vicinity.

The drilling by Gays River Lead Mines Limited outlined 4 zones of mineralization in an area about 400 m by 900 m. Over 800,000 t of mineralized Windsor Group carbonate (galena, sphalerite, pyrite, marcasite, and chalcopyrite) were defined overlying and flanking a northeast-trending anticlinal Meguma greywacke basement high. Grades for the 4 zones ranged from 1.10% to 3.50% combined Pb plus Zn, with an average of 2.32% combined Pb plus Zn. Most, if not all, assays were from sludge samples.

9.1.2 Timeline: 1962

Gunnex Limited carried out extensive soil sampling in the Gays River area in 1962. Anomalies were encountered only over areas of previously known mineralization where overburden was thin. An induced polarization survey indicated only a very weak response over known mineralization and did not add any new target areas. The lack of encouraging response on the periphery of the earlier defined mineralized area prompted Gunnex to forego any further exploration activity.

9.1.3 Timeline: 1968 – 1969

In 1968 and 1969, Penarroya Canada Limited completed extensive soil sampling and geological mapping in the Gays River and Meaghers Grant areas. Two diamond drill holes in the Meaghers Grant area intersected minor Zn mineralization. No drilling was carried out in the Gays River area even though a number of soil anomalies had been identified. Most of the major anomalies corresponded with previously known mineralization. However, two new anomalous areas were defined. They occur near Carroll's Corner and in the Black Brook area east of the Gays River, and define a northeast trending geochemical high. The latter area is close to the northeast end of the presently defined Gays River Deposit itself.

9.1.4 Timeline: 1971

Texasgulf Inc. drilled four diamond drill holes in the Gays River area in 1971. One hole adjacent to a Gays River Lead Mines drill hole confirmed significant mineralization in the carbonates. The remaining holes tested one soil anomaly southeast of Gays River and two areas northwest of Gays River. No encouraging mineralization or carbonate build-ups were intersected in the last three holes and work was terminated.

9.1.5 Timeline: 1972 - 1984

In 1972 personnel of Cuvier prospected the Gays River area and located significant mineralized float material to the south of the old occurrence (MacEachern and Hannon, 1974) and subsequently acquired the ground. Cuvier also outlined geophysical and geochemical anomalies. In September of 1972, Cuvier optioned the property to Esso, with Esso holding a 60% interest and acting as the operator. Cuvier formed a joint venture with Preussag Canada Ltd. ("Preussag") to finance Cuvier's 40% interest in the property.

Both Cuvier and Esso were of the opinion that the area had the proper geological setting for a Mississippi Valley-type deposit. Esso recognized the possible existence of a reef complex trending northeasterly from the old Gays River drilling site. The source of the mineralized boulders had not been located and a combination of deep glacial till and lack of outcrop would necessitate fence-type drilling in geologically favourable areas for the purpose of obtaining geological information, as well as locating any mineralized areas.

A total of 20 holes were drilled prior to drilling the discovery hole 2.5 km northeast of the original showing along the postulated reef trend. The discovery hole intersected 3.35 m averaging 7% Zn (MacEachern and Hannon, 1974).

From October 1972 to August 1974, Esso / Cuvier drilled off the deposit and identified 12 million tonnes averaging 7% Zn plus Pb (Patterson, 1993) over an area of approximately 4 km by 220 m at depths ranging from 20 m to 200 m (450 surface core holes)[1].

The initial mine by Esso began with developing the exploration decline in 1976 across the central portion of the mineralized zone to verify mining conditions, the grade, and continuity of the mineralization and to provide bulk samples for metallurgical testing. The decline was 760 m in length but by mid-1979 some 1,800 m of drifting and 744 m of underground development had been completed. The deepest workings were at a vertical depth of 100 m. In December of 1977, Esso purchased Cuvier's and Preussag's interests in the property and formed Canada Wide Mines to develop and mine the deposit.

During the next 2 years, various feasibility studies were carried out. Recoverable proven plus probable reserves were then estimated at 4.7 million tonnes at 2.8% Pb and 4.2% Zn (WMC, 1995). Esso commenced with the construction of the mill and other facilities in August 1977. The 1,350 t processing plant was commissioned in October 1979 and the mine was further developed to support a 1,350 t per day operation.

From 1978 until 1981, Esso operated the mine and targeted the lower grade mineralization using a trackless, lower cost, bulk Room and Pillar mining method approach. The higher-grade mineralization near the carbonate contact was not part of the mine plan. Operations continued until August 1981, when production was suspended, with the exception of an underhand Cut-and-Fill technique test stope. Mining conditions exacerbated by bad ground conditions and excessive water inflow caused the operation to be suspended. During the operation, a total of 553,688 t of mineralized material averaging 1.36% Pb and 2.12% Zn were produced and run through the mill. The operation also removed 272,000 t of waste. Throughout this period efforts to achieve the full production rate, as well as efforts to mine areas

[1] A summary table of all known drilling at the Gays River Deposit by all exploration companies over the years is included as Table 10-1.

of higher-grade mineralization, were complicated by the combination of the complex geological setting and the severe hydrological problems.

The plant was shut down in 1982 as a result of operating losses due to lower than expected grades, higher than expected operating costs, the difficult water problems, and low metal prices.

Seabright acquired the mine and mill in 1984, but despite a favourable feasibility study, did not reactivate the mine due to depressed metal prices at the time.

9.1.6 Timeline: 1985 - 1987

Seabright's primary intention was the usage of the mill facility to process gold ore from their outlying properties, and a secondary intent to later re-open the Gays River mine (WMC, 1995). At the time, Seabright was mining (bulk sampling) gold-bearing quartz veins from four small operations: Beaver Dam, Forest Hill, Caribou, and Moose River, all located within the Meguma Group (Cambro-Ordovician).

The milling facility was converted for gold processing. The mine was not re-opened by Seabright at that time, as a sharp drop in Zn prices rendered the underground mining operation uneconomic.

9.1.7 Timeline: 1987 - 1991

In 1988, Westminer Canada Limited (WMC) purchased Seabright. A review of the deposit, including the drilling of 89 surface core holes, led WMC to a positive production decision based on a reinterpretation of the geology and mining method. They began dewatering the underground workings in early 1989. Following the success of the mine dewatering and a test mining period to assess the suitability of the proposed Narrow Vein Cut-and-Fill mining method to extract the higher-grade resource zones, the mine was placed back into production. It reached commercial production rates in March 1990 (WMC, 1995) at a rate of 800 t/d.

WMC's initial approach was to drive small 2.5 m by 2.5 m Cut-and-Fill stopes adjacent to the trench material. Dry waste rock backfill was placed after each lift. In most areas, the method allowed the higher-grade resource on the carbonate-trench contact to be extracted. In one area, WMC successfully tested the Room and Pillar mining method (Nesbitt Thomson, 1991). A total of 187,010 t of ore at an average grade of 3.5% Pb and 7.47% Zn were mined during WMC's involvement on the property.

Hydrological difficulties causing poor ground conditions continued to play a factor in the mine operation. In May 1991, rising water levels due to the spring runoff forced the cessation of mining in a number of stopes and WMC decided to place the mine in project mode. Following the suspension of production in 1991, WMC carried out an extensive program to understand the mine hydrology and concluded that the groundwater could be successfully managed so that mining operations would no longer be adversely affected.

WMC has identified the Eastern zone of the deposit as an area for possible early development. This is because ground conditions are substantially better due to the hanging wall generally comprising gypsum / anhydrite rather than trench. The grade is also higher relative to other sections of the deposit. The Eastern area appears promising for additional resources.

WMC thoroughly assessed the property in 1991 and prepared a revised mine plan to resume mine production. The revised plan provided for more mechanization of the mining method, institution of paste backfill, increased groundwater drainage through screened drainage wells, and a revised pumping system. However, the operation was WMC's only Zn and Pb producer, was not associated with any downstream smelting facilities, and was a smaller operation relative to other corporate assets. For these reasons, the property did not fit within WMC's corporate strategy to focus on large-scale operations and for this reason the property was sold to Savage Resources.

9.1.8 Timeline: 1996 - 1999

After acquiring the Scotia Mine in 1996, Savage conducted two exploration drilling programs to fill in the gaps from prior drilling and improve the mineral resource estimate on the mine property. In December 1996, 36 diamond drill holes, totaling 1,325 m were drilled in the central mine area adjacent to the underground mine entrance to test the continuity of the disseminated low grade mineralization in the back reef (known as the sand pit area; an area of commercial aggregate). In April and May 1997, an additional 30 diamond drill holes totaling 2,339 m were drilled in the Northeast Zone (as identified by WMC). Both programs were successful and confirmed the presence of low grade mineralization (in the central area) and higher-grade mineralization (in the Northeast Zone). According to Cullen (1997), the results of the drilling (based on a 7% Zn-equivalent cut-off grade) enhanced some areas of the Northeast Zone and diminished other areas. He also states that a complete revision of some of this area (with additional drilling evaluation) be completed prior to any production decision.

Savage dewatered the underground workings from June to August 1997, and started to rehabilitate the mine before a decision was made to extract the ore in the main, central zone using Open Pit methods. An open pit design was prepared using appropriate technical criteria for ore mining and waste stripping (Gemcom and Whittle 3-D Optimization). The preliminary mine plan assumed the processing of 1,350 t/d with the ore coming from a combination of underground (1,000 t/d) and open pit operations (350 t/d).

In early 1999, ownership of Savage was transferred to the Australian mining company Pasminco Canada Limited (Pasminco).

9.1.9 Timeline: 2001 – 2003

Regal Mines Limited (Regal Mines) purchased Pasminco Resources Canada Company (Pasminco Resources) and its assets in February 2002. Regal was owned 50% by OntZinc Corporation (OntZinc) and 50% by Regal Consolidated Ventures Limited (Regal Consolidated). As part of the sale, Pasminco Canada Holdings Inc. (Pasminco Holdings) retained a 2% net smelter return (NSR) royalty on future production. OntZinc acquired Regal Consolidated's 50% interest in December 2002 to own 100% of Pasminco Resources. Savage Resources Limited is the successor of Pasminco Holdings and currently holds the 2% royalty. Pasminco Resources was later renamed ScoZinc Limited (ScoZinc). The mining and environmental permits are still in force and are held by ScoZinc along with all the Scotia Mine assets.

9.1.10 Timeline: 2004 – 2006

Exploration activity by ScoZinc included diamond core drilling, a hydraulic mining test, prospecting of the general area, geological compilation of past relevant data, and two lines (ten samples) of Mobile Metal Ion Geochemistry (MMI) across areas of known mineralization covered by thick accumulations of glacial till. The results of the MMI survey were inconclusive.

A hydraulic mining test was performed to determine whether such a method might be useful to uncover the glacial overburden and some of the trench material in the area of the low grade, potentially surface mineable resources. This was primarily performed near the area of the sand pit next to the original portal. The test showed that it is possible to mine the sandy overburden in the current pit bottom using dredging methods.

Six holes were drilled through the trench unit using a soil drilling rig. The trench is a geological unit that occurs between the gypsum and dolomite units. The purpose of this program was to characterize the soils that make up the trench. Four holes were drilled in the Central Zone near the current pit. The two other holes were drilled near the highway (Hwy 224) in the East Zone.

The soil holes in the Central Zone around the current pit consisted mainly of dark brown clay with fine- to medium-grained sand. Rock fragments, rounded-to-angular, were occasionally noted. The soil holes in the East Zone near the river and highway consisted of fine- to medium-grained sand with minor clay. This observation may be an important factor during future mining. Permeability underneath the river is expected to be high to a depth of at least 20-30 m. This will adversely affect slope stability should the walls of an open pit approach the river.

Twenty-five diamond core drill holes (1,845.3 m) were completed by ScoZinc on the Scotia Mine property. Seventeen of these holes were meant to further define the Zn / Pb mineralization contained within the reef carbonate, while the remaining eight holes were meant to test the gypsum potential immediately overlying the mineralized zones.

Four holes (477 m) were completed in the northeastern portion of the deposit, while thirteen holes (1,172 m) were completed in the central area of possible lower grade open pit mineralization. The program was moderately successful in the central area with Zn values consistently in the 2% to 4% range over 1 m to 2 m (Table 10-1). The drilling program in the northeastern zone proved less successful with mineralized intervals being quite thin.

Four holes (673.3 m) were drilled in the Northeast Zone and an additional four in the central area to test the overlying gypsum in the hanging wall of the base metal mineralization. The holes were drilled to obtain core samples of the gypsum deposits that immediately overlie the mineralized zones. The purpose of the samples was to carry out early tests of gypsum consistency and quality, as well as to confirm preliminary estimates of the probable size of the gypsum resource adjacent to the mineralized trend.

In most of the diamond drill holes, a gypsum "cap," 20–30 m thick was encountered. Grade was highest (greater than 90% gypsum) near the bedrock surface and decreased with depth. At 20–30 m depth, gypsum grade dropped below 80%, transitioning to anhydrite over an interval of approximately 10 m. Because the gypsum was quite hard, it was difficult to visually determine the contact between gypsum and anhydrite.

9.1.11 Timeline: 2007 – 2008

ScoZinc began surface mining the deposit in 2007 and carried on into 2008. Due to a drastic fall in metal prices, ScoZinc placed the mine on care and maintenance status.

In 2008, ScoZinc drilled 17 diamond drill holes through the Northeast Zone (refer to Section 10.0).

9.1.12 Timeline: 2011

Selwyn drilled a further 39 drill holes, totaling 4,950.5 m between 11 August and 11 October 2011 (see Section 10.2.2).

9.1.13 Timeline: 2012

A helicopter-borne geophysical survey over an area including the Northeast Zone was carried out in February, 2012. The helicopter flew over 55.9 line-kilometers over the study block, with prinicipal geophysical sensors being a versatile time domain electromagnetic system, and a horizontal magnetic gradiometer. Results included the identification of a large-scale conductive zone that covered the northern quarter of the block, as well as several localized magnetic anomalies in close proximity of the Northeast Zone, thereby representing exploration targets.

10.0 DRILLING

10.1 Sample Length – True Width Relationship

The sample intervals do not necessarily represent true widths. The orientation of the deposit is variable, meaning the true width of any given intercept must be calculated with reference to the geological model. The orientation of the deposit is well known and is described in Section 7.2.

10.2 Gays River Deposit

To date, 1,419 diamond core drill holes have been drilled on the Gays River Deposit (refer to Figure 10-1 and Table 10-1). The majority were drilled to determine the characteristics of the Zn- and Pb-mineralized dolomite.

ScoZinc drilled 17 holes, totaling 1,613.5 m, through the Northeast Zone in 2008. These collars, as well as the collars from ScoZinc's 2004 program, are shown in magenta in Figure 10-1.

Selwyn drilled a further 39 drill holes totaling 4,950.5 m between 11 August 2011 and 11 October 2011 (see Section 10.2.2).

Most of the 914 surface holes were drilled vertically. The azimuth and dip of the 467 holes drilled from the underground workings was variable.

Generally, holes were drilled to fully penetrate the dolomite reef and continue on until no additional mineralization was found. This resulted in most drill holes being drilled a few meters beyond the dolomite reef.

A compilation of core logs and sample assays from the 2008 program is given in the updated mineral resource report NI 43-101 filed 08 October 2012. Historical logs are provided in the previous technical report for the property (MineTech, 2006).

Table 10-1: Historical Surface and Underground Diamond Drilling Activity[2]

From	To	Holes with Info[3]	Meters	Time Frame	Company
Surface Holes					
1	72	70	2,951.7	1951-1952	Gays River Lead Mines
73	740	646	59,123.6	1972-1982	Imperial Oil/Canada Wide Mines
741	900	89	7,596.8	1985-1995	Seabright, then Westminer (undifferentiated)
901	966	66	3,664.0	1997	Savage/Pasminco
967	991	25	1,864.3	2004	ScoZinc
1130-08	1146-08	17	1,613.5	2008	ScoZinc
MNZ-001	MNZ-039	39	4950.5	2011	Selwyn
Subtotal		**952**	**81,764.4**		
Underground Holes					
1	341	318	7,460.7	1979-1982	Imperial Oil/Canada Wide Mines
342	651	149	4,434.9	1985-1995	Seabright, then Westminer (undifferentiated)
Subtotal		**467**	**11,895.6**		
Total		**1,419**	**93,660**		

10.2.1 Sample Statistics

Sample statistics were calculated for sampling within the carbonate. All samples for which at least one metal (Zn or Pb) was assayed were considered. Most samples were assayed for both Zn and Pb. Depending on the amount of visible mineral, some samples were assayed for only one metal. The total sample count was 8,022.

The samples from the 2011 drill program were not included in the sample statistics calculations.

The mean sample interval length was 1.44 m with a standard deviation of 0.82 m (see Table 10-2). Skewness is a measure of symmetry, or more precisely, the lack of symmetry. The positive value for skewness indicates that the data is skewed right, meaning that the right tail is heavier than the left tail. This is also shown in the histogram in Figure 10-1. The aggregate sample length was 11,522 m.

[2] Data supplied by ScoZinc.

[3] The electronic database does not contain information for underground holes 342 – 499.

The mean Zn grade was 3.55%. From the histogram, we can see that Zn assays are approximately lognormal. The range in Zn content was 0% to 62.10%. Theoretically, the maximum possible Zn assay is 67.10%—the Zn content of pure sphalerite.

The mean Pb grade was 1.91%. From the histogram, we can see that Pb assays are also approximately lognormal. The range in Pb content was 0% to 79.50%. Theoretically, the maximum possible Pb assay is 86.6%—the Pb content of pure galena.

Sample statistics are further examined in Section 14.0.

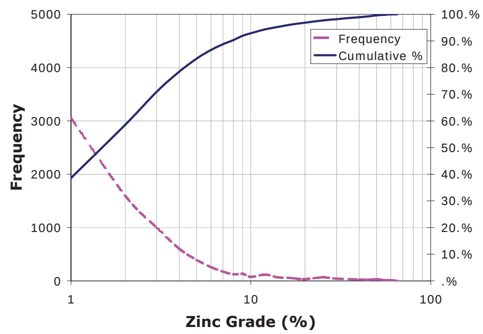
Table 10-2: Descriptive Statistics

Descriptive Statistic	Zn Grade (%)	Pb Grade (%)
Mean	3.55	1.91
Standard Error	0.08	0.07
Median	1.52	0.12
Mode	0.02	0.01
Standard Deviation	6.79	6.24
Sample Variance	46.17	38.99
Kurtosis	25.17	52.86
Skewness	4.60	6.56
Range	62.10	79.50
Minimum	0.00	0.00
Maximum	62.10	79.50
Sum	n/a	n/a
Count	8,022	8,022

Figure 10-1: Sample Histograms

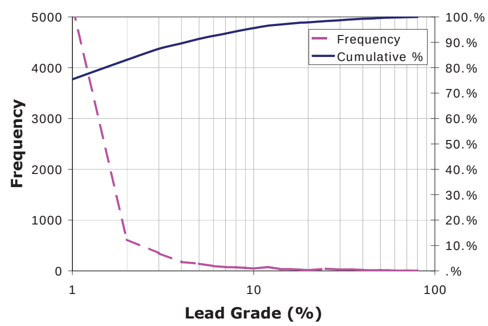
Zinc Histogram

Range	Frequency	Cumulative %
0	39	.49%
0-1	3056	38.58%
1-2	1599	58.51%
2-3	999	70.97%
3-4	603	78.48%
4-5	393	83.38%
5-6	259	86.61%
6-7	174	88.78%
7-8	123	90.31%
8-9	132	91.96%
9-10	74	92.88%
10-12	118	94.35%
12-14	68	95.20%
14-16	58	95.92%
16-18	42	96.45%
18-20	33	96.86%
20-25	65	97.67%
25-30	41	98.18%
30-35	33	98.59%
35-40	27	98.93%
40-45	23	99.21%
45-50	32	99.61%
50-55	15	99.80%
55-60	14	99.98%
60-65	2	100.00%
65+	0	100.00%



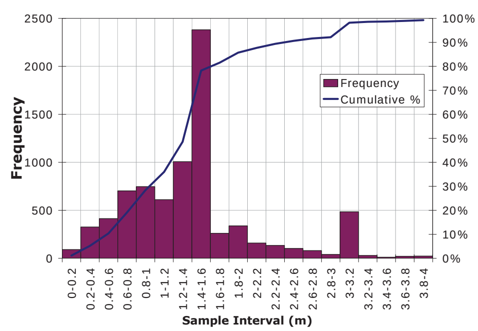
Lead Histogram

Range	Frequency	Cumulative %
0	851	10.61%
0-1	5200	75.43%
1-2	616	83.11%
2-3	347	87.43%
3-4	173	89.59%
4-5	141	91.35%
5-6	94	92.52%
6-7	73	93.43%
7-8	70	94.30%
8-9	59	95.04%
9-10	45	95.60%
10-12	77	96.56%
12-14	34	96.98%
14-16	33	97.39%
16-18	25	97.71%
18-20	12	97.86%
20-25	41	98.37%
25-30	26	98.69%
30-35	29	99.05%
35-40	17	99.26%
40-45	10	99.39%
45-50	15	99.58%
50-55	10	99.70%
55-60	7	99.79%
60-65	3	99.83%
65-70	7	99.91%
70-75	3	99.95%
75-80	4	100.00%
80+	0	100.00%



Sample Interval Histogram

Range	Frequency	Cumulative %
0	0	.00%
0-0.2	91	1.13%
0.2-0.4	326	5.20%
0.4-0.6	413	10.35%
0.6-0.8	702	19.10%
0.8-1	746	28.40%
1-1.2	611	36.01%
1.2-1.4	1007	48.57%
1.4-1.6	2381	78.25%
1.6-1.8	260	81.49%
1.8-2	338	85.70%
2-2.2	159	87.68%
2.2-2.4	134	89.35%
2.4-2.6	103	90.64%
2.6-2.8	80	91.64%
2.8-3	40	92.13%
3-3.2	485	98.18%
3.2-3.4	29	98.54%
3.4-3.6	11	98.68%
3.6-3.8	21	98.94%
3.8-4	23	99.23%
4+	62	100.00%



10.2.2 Gays River Drilling, 2011

10.2.2.1 Type and Extent of Drilling

Selwyn drilled a further 39 drill holes totaling 4,950.50 m between 11 August 2011 and 11 October 2011. Of the 39 holes drilled, 34 were drilled on Mineral Lease 10-1 and five were drilled on Exploration License 6959. Three of the 39 holes were drilled to the northeast of the existing pit, while the remaining 36 were drilled in a broad area to the southwest of the pit. The deepest hole was 195 m deep, the shallowest was 43 m, and the mean depth was 128 m. Drilling was carried out by Logan Drilling Group of Stewiacke, Nova Scotia.

Drill holes were planned to target Zn / Pb sulfide mineralization that possessed the potential to expand upon the current mineral resource or provide greater definition. Targets were primarily chosen to the southwest of the current mine pit, along the margins of, and clustered toward the southwest extent of the Main Zone of the deposit.

10.2.2.2 Drilling Procedures

Once targets were determined, drill collar locations were calculated using a projected drill hole inclination that would intersect the Gays River Formation carbonate bank front at an angle closest to perpendicular. Targets were fine-tuned based on ground factors, including terrain, proximity to watercourses, and property boundaries.

Drilling was carried out under the direction of ScoZinc and Selwyn exploration staff. A skid-mounted Longyear-38 diamond drill was used to complete all drill holes, and was dragged onto each drill pad with the assistance of a small John Deere bulldozer. In addition to the drill, a covered water pump and drill rod sloop were also dragged to the area by bulldozer.

All recovered core was boxed, lidded, and returned to the ScoZinc core shack where it was logged and sampled by ScoZinc exploration staff.

All drill core was logged, cut, and sampled by ScoZinc and Selwyn staff at the ScoZinc core shack. Both geotechnical and geological data was collected from all drill core. Geotechnical data collected included total core recovery, rock quality designation, strength and weathering data, Q System discontinuity orientation data, and rock mass rating system data. Geological data collected included stratigraphic contacts, and all lithological, mineralogical, and structural observations of note.

10.2.2.3 Sampling

Thirty-eight drill holes intersected the Gays River Formation). Silver and base metal analyses were conducted by a 23-element, four-acid digestion, ore-grade ICP-AES technique. Drill hole MNZ-005 was not sampled, as it did not intersect the formation.

A total of 722 samples were submitted to Acme Analytical Laboratories (Acme) in Vancouver, British Columbia. Of those 722 samples, 559 samples (77.4%) were actual core samples and 163 samples (22.6%) were QA/QC samples (see Section 11.3).

10.2.2.4 Summary and Interpretation of Results

All but one drill hole (MNZ-005) successfully intersected the mineralized Gays River Formation, although thicknesses and grades were somewhat variable.

10.2.2.5 Drill Hole Collar Data

See Table 10-3 for drill hole collar data.

10.2.3 Logistics of Acadian Drill Program

Logan Drilling of Stewiacke, Nova Scotia, was contracted to complete 2007–2008 drilling utilizing skid-mounted Longyear-38 drilling equipment equipped to recover NQ sized drill core (4.76 cm diameter). One drill was typically employed, but a second drill was periodically on site. Both machines typically operated 24 hours per day. Mercator was contracted to manage day-to-day drilling operations and provided on-site supervision, transportation of core to the secure logging facility at Acadian's Scotia Mine, plus logging of drill core and supervision of core sampling services. A registered land surveyor surveyed drill hole collars, and all drill holes were coordinated to the local Scotia Mine grid system.

Table 10-3: Drill Hole Collar Data

Hole ID	UTM Easting	UTM Northing	Elev. (m)	Az (true)	Dip	EOH (m)
MNZ-001	472840.32	4986616.85	23.80	154.0	-75.0	165.1
MNZ-002	472817.22	4986556.71	32.51	155.0	-75.0	194.9
MNZ-003	472745.12	4986528.99	37.54	152.0	-65.0	161.0
MNZ-004	472705.01	4986537.07	39.28	155.0	-60.0	152.0
MNZ-005	472909.24	4986221.07	44.15	160.0	-87.0	56.0
MNZ-006	472581.14	4986477.00	45.45	158.0	-70.0	176.0
MNZ-007	472714.91	4986245.28	46.25	158.0	-80.0	63.0
MNZ-008	472548.94	4986453.18	46.78	143.0	-70.0	194.0
MNZ-009	472577.79	4986408.98	45.01	155.0	-70.0	167.0
MNZ-010	472668.51	4986234.87	47.74	152.0	-80.0	92.3
MNZ-011	472467.67	4986438.44	42.80	157.0	-70.0	179.0
MNZ-012	472492.13	4986422.43	43.95	150.0	-50.0	165.0
MNZ-013	472593.28	4986232.83	51.07	158.0	-75.0	80.0
MNZ-014	472460.65	4986399.30	45.53	152.0	-60.0	147.5
MNZ-015	472403.74	4986408.57	45.69	154.0	-65.0	143.0
MNZ-016	472334.97	4986387.29	44.59	146.0	-65.0	161.8
MNZ-017	472317.75	4986327.62	46.19	140.0	-75.0	187.0
MNZ-018	472435.54	4986153.22	51.11	150.0	-62.0	101.0
MNZ-019	472248.00	4986285.53	42.00	160.0	-81.0	164.0
MNZ-020	472347.33	4986078.11	52.03	151.0	-80.0	65.0
MNZ-021	472173.21	4986281.68	37.36	152.0	-66.0	155.0
MNZ-022	472077.24	4986272.60	20.88	150.0	-70.0	135.0
MNZ-023	472207.90	4986094.90	46.49	148.0	-80.0	74.0
MNZ-024	472238.97	4986038.08	47.46	134.0	-87.0	68.0
MNZ-025	472310.88	4985911.89	46.89	150.0	-87.0	43.2
MNZ-026	472087.89	4986245.72	26.58	150.0	-52.0	136.0
MNZ-027	472135.85	4986130.58	40.68	150.0	-60.0	84.0
MNZ-028	472044.77	4986205.24	20.97	338.0	-79.0	140.0
MNZ-029	472047.95	4986148.28	27.68	152.0	-51.0	95.0
MNZ-030	472099.70	4986061.13	38.52	163.0	-86.0	74.4
MNZ-031	472044.76	4986202.24	20.97	237.0	-65.0	136.0
MNZ-032	472125.32	4985935.07	53.32	323.0	-79.0	116.0
MNZ-033	472078.81	4985964.98	48.72	217.0	-76.0	143.0
MNZ-034	471955.78	4986084.56	31.66	150.0	-83.0	106.0
MNZ-035	471937.90	4986048.18	33.99	147.0	-65.0	122.7
MNZ-036	471954.85	4986081.72	31.36	276.0	-70.0	164.0
MNZ-037	473401.79	4986905.02	18.47	145.0	-86.0	191.0
MNZ-038	473488.02	4986897.65	19.42	126.0	-64.0	133.9
MNZ-039	473667.10	4986733.54	30.92	170.0	-87.0	53.0

11.0 SAMPLING PREPARATION, ANALYSIS, AND SECURITY

11.1 Gays River Deposit (pre-2008)

There is no written record regarding the sampling method employed during the early exploration years (i.e., pre-1970s) in the Scotia Mine area.

The exploration approach and sample collection procedures employed by the more recent exploration efforts reflects thorough sampling methodology and documentation procedures. Exploration activity was carried out in a professional manner by a team of local, experienced geologists and technicians supervised by Esso's, Seabright's, Westminer's, Savage's, and ScoZinc's professional staff. The work has been well organized throughout their exploration efforts. More recently, computer facilities were available to generate reports and prepare maps, etc., from the vast database.

The assay data and other parameters for all core drilling programs and underground work were entered into a computerized database using Microsoft Excel and resource estimate generating software programs. The quality control and validation of the coded data included steps to ensure that the assay intervals and the sample locations were correct. To ensure accuracy of the database, all assays were coded and the data entry system automatically checked for interval overlaps. The coded assays were also printed and a visual inspection was completed for comparison with the original (logged) data sheets. The sample locations were validated with appropriate plotting and visual checks against the original sections and plans.

Core drilling was carried out using North American service providers with the collection of BQ and NQ core. The portions of core to be analyzed were either split or sawed into two sections with one half submitted for analysis, the other half remaining in the core tray. All sampling procedures were carried out on site.

Sampled core lengths were determined visually. All drill holes were logged, noting lithology, structure, alteration, and mineralization. Core recovery was generally greater than 90%. Early in the exploration program, the samples were sent via air cargo to several analytical laboratories; however, after the construction of the mill facility, the internal laboratory was used.

Core samples from Savage's 1997 drilling program and ScoZinc's 2004 drilling program were submitted to the Minerals Engineering Centre of Dalhousie University (formerly Technical University of Nova Scotia) in Halifax. The laboratory is independent of Savage, ScoZinc, and Selwyn. The laboratory is not International Standard Organization (ISO) accredited.

According to the Minerals Engineering Centre, the core samples were prepared using the following process:

1. Dried and then crushed in one or more jaw crushers, depending on the original size, to under one-quarter inch.
2. Split in a Jones riffle to a mass of 150–200 g.
3. Pulverized using a ring and puck pulverizer to 80% minus 200 mesh (75 microns).
4. Put into either a bag or a vial. Rejects were kept for 6 months.

The sample analysis procedure consisted of the following:

1. Sample lots of 1 g were digested with hydrochloric-nitric-hydrofluoric-perchloric acids.
2. Elements were determined by Flame Atomic Absorption, with a detection limit of 1 ppm.
3. Arsenic was determined by atomic absorption/hydride generation method.

Reference standards from CANMET were routinely used as internal checks on the accuracy of the analysis.

11.2 Gays River Deposits (2008)

11.2.1 Site Procedures

Cullen (2011) provided the following description for the sampling methods that were used for the 2008 drilling program:

> ***Sample Security and Chain of Custody***
> *In accordance with the sample protocol established by Mercator for the 2008 drilling program, all drill core was delivered from the drill site to the secure and private core logging facility at Acadian's Scotia Mine by either Logan Drilling Limited staff or Mercator field staff. Drill core logging was carried out by a Mercator geologist who also marked core for sampling and supervised core splitting by a technician using a rock saw. Sample tag numbers from a three-tag sample book system were used for the program, with one tag showing corresponding down hole sample interval information placed in the sampled core boxes at appropriate locations, one tag lacking down hole interval information placed in the core sample bag for shipment to the laboratory, and the third tag with sample interval information retained in the master sample book for future reference and database entry purposes. After sampling, core boxes were closed and placed in storage at the Scotia Mine site. Sealed sample bags were placed in an ordered sequence prior to insertion of quality control samples,*

preparation of sample shipment documentation, checking, and placement in plastic buckets for shipment by commercial courier to Eastern Analytical Limited ("Eastern"), a recognized commercial laboratory located in Springdale, Newfoundland. A check pulp sample split was prepared at Eastern for every 25th submitted sample and these were labelled, placed in a sealed envelope and returned to Mercator. After insertion of certified standard and blank samples, all check samples were sent to ALS Chemex in Sudbury, ON for independent analysis of zinc and lead levels. All other prepared pulps and coarse reject material was stored at Eastern until the end of the program, at which time they were shipped back to Scotia Mine for secure archival storage.

11.2.2 Laboratory Procedures

Cullen (2011) provided the following description for the sampling methods that were used for the 2008 drilling program:

Core Sample Preparation

Core samples received by Eastern were organized and labelled and then placed in drying ovens until completely dry. Dried samples were crushed in a Rhino Jaw Crusher to consist of approximately 75% minus 10 mesh material. The crushed sample was riffle split until 250 to 300 grams of material was separated and the remainder of the sample was bagged and stored as coarse reject. The 250 – 300 gram split was pulverized using a ring mill to consist of approximately 98% minus 150 mesh material. All samples underwent ICP analysis, for which a 0.50g portion of the pulverized material was required. Those samples containing greater than 2200 ppm of zinc or lead were then processed using ore grade analysis for which 0.20g of pulverized material was required. Laboratory sample preparation equipment was thoroughly cleaned between samples in accordance with standard laboratory practice.

Check sample splits of pulverized core were submitted to the ALS Chemex laboratory facility in Sudbury, Ontario as part of the project quality control and assurance protocol. This material was prepared in approximately 100 gram bagged splits by Eastern and returned to Mercator for subsequent submission to ALS Chemex. Since the received split material had already been pulverized, further preparation was limited to homogenization and splitting of a 0.4 g portion for subsequent analysis.

Core Sample Analysis

Eastern Analytical procedures outlined below pertain to all core samples from the ¬2008 drill program.

ICP Analysis: A 0.50 gram sample is digested with 2ml HNO3 in a 95oC water bath for ½ hour, after which 1ml HCL is added and the sample is returned to the water bath for an additional ½ hour. After cooling, samples are diluted to 10ml with de¬ionized water, stirred and let stand for 1 hour to allow precipitate to settle.

For ore grade analysis base metals (lead, zinc, copper), a 0.20g sample is digested in a beaker with 10ml of nitric acid and 5ml of hydrochloric acid for 45 minutes. Samples are then transferred to 100ml volumetric flasks and analyzed on the Atomic Absorption Spectro-Photometer (AA). The lower detection limit is 0.01% and the upper detection limit is >2200 ppm lead or zinc.

For silver, a 1000 mg sample is digested in a 500ml beaker with 10ml of hydrochloric acid and 10ml of nitric acid with the cover left on for 1 hour. Covers are then removed and the liquid is allowed to evaporate leaving a moist paste. 25ml of hydrochloric acid and 25ml of deionized water are then added and the solution is gently heated and swirled to dissolve the solids. The cooled material is transferred to 100ml volumetric flask and is analyzed using AA. The lower detection limit is 0.01oz/t of silver with no upper detection limit.

A prepared sample is digested in 75% aqua regia for 120 minutes. After cooling, the resulting solution is diluted to volume (100 ml) with de-ionized water, mixed and then analyzed by inductively coupled plasma - atomic emission spectrometry or by atomic absorption spectrometry.

11.3 Gays River Deposit (2011)

11.3.1 Site Procedures

All drill core was logged, cut, and sampled by ScoZinc and Selwyn staff at the ScoZinc core shack. Sampling of mineralized core from the Gays River Formation and adjacent units involved breaking the mineralized range into 20–150 cm samples, inserting regular QA/QC duplicate, blank, and standard samples as per company protocol, and halving each sample longitudinally with a diamond bladed rock saw. One half of the sample was placed back in the core box for storage, and the other half was bagged and sent away for assay in Vancouver.

11.3.2 Laboratory Procedures

Samples were assayed at Acme for preparation and analysis. The Acme laboratory in Vancouver is certified ISO9001:2008 compliant for the provision of assays and geochemical assays. Acme is independent of the issuer.

Samples were weighed, analyzed using four-acid digestion multi-element ICP-ES (method 7TD), and tested for specific gravity (method G8SG).

The general sample preparation method used by Acme for rock and drill core is described as follows:

> *Rock and Drill Core crushed to 80% passing 10 mesh (2 mm), homogenized, riffle split (250g, 500g, or 1000g subsample) and pulverized to 85% passing 200 mesh (75 microns). Crusher and pulveriser are cleaned by brush and compressed air between routine samples. Granite/Quartz wash scours equipment after high-grade samples, between changes in rock colour and at end of each file. Granite/Quartz is crushed and pulverized as first sample in sequence and carried through to analysis.*

Method 7TD is described by Acme as follows:

> *0.5g sample split is digested to complete dryness with an acid solution of H2O-HF-HClO4-HNO3. 50% HCl is added to the residue and heated using a mixing hot block. After cooling the solutions are made up to volume with dilute HCl in class A volumetric flasks. Sample split of 0.1g may be necessary for very high-grade samples to accommodate analysis up to 100% upper limit.*

Method G8SG is described by Acme as follows:

> ***G812 Specific Gravity Pulp, SG:***
> *A split of dry pulp is weighed to a class A volumetric flask. Flask and pulp are weighed precisely on a top-loading balance. Measure and record the weight then calculate for specific gravity.*
>
> ***G813 Specific Gravity Core, SG:***
> *Analysis can be conducted on whole samples of rock or core in irregular shape. Specific gravity is determined by measuring the displacement of water. A sample is dried at 105°C to remove all moisture then allowed to cool. The sample of the rock or drill core is first weighed in air then submerged in a container of water. Measure the mass of immersed sample and record the weight then calculate for specific gravity. Sample can also be coated with a thin layer of hot wax so that any soluble material in the core or rock is not in contact with the water.*

11.3.3 Quality Control Procedures

Quality Control Samples

Of the 722 samples sent to Acme, 51 were standards, 58 were duplicates, 54 were blanks, for a total of 163 QA/QC samples. The remaining 559 were regular assays.

Of the blanks, all but one were at the lower detection limit for Pb (0.01%), while a single sample was above the lower detection limit, with a value of 0.02% Pb. Similarly, all but three of the blanks were at the lower detection limit for Zn (0.005 %), while three samples were above the lower detection limit, with values of 0.01%, 0.02%, and 0.04% Zn.

Of the duplicates, 38 of the 58 had a difference in Pb at or below the detection limit. For the remaining samples, the average difference was 0.24% Pb; 9 samples had a difference at or above 0.20% Pb, with the greatest difference being 0.91% Pb.

Of the 58 duplicates, 24 had a difference in Zn at or below the detection limit. For the remaining samples, the average difference was 0.19% Zn; 9 samples had a difference at or above 0.20% Zn, with the greatest difference being 0.95% Zn.

Two types of standard were used: Standard F (28 used) and Standard G (23 used). Both were created by WCM Sales Ltd. Standard F has a mean value of 1.240% Pb and 2.000% Zn, while Standard G has a mean value of 6.680% Pb and 3.780% Zn, both with a tolerance of +/- 2 standard deviations. Table 11-1 summarizes the results.

Table 11-1: 2011 Sampling Standards

Standard	Expected Value	Average Tested Value	Minimum Tested Value	Maximum Tested Value
Standard F – Pb	1.240%	1.21%	1.14%	1.28%
Standard F – Zn	2.000%	2.13%	2.02%	2.22%
Standard G – Pb	6.680%	6.55%	6.20%	7.11%
Standard G – Zn	3.780%	3.91%	3.76%	4.06%

Results from the check samples are within acceptable limits.

Umpire Assays

Split pulps of 135 samples were re-analyzed at the ALS Minerals laboratory in Vancouver (ALS). ALS Minerals is a division of ALS Ltd., and is independent of the issuer and is certified to the ISO/IEC 17025:2005 by the Standards Council of Canada (SCC).

The comparison found that the vast majority of the split pulps are within a +/-15% tolerance. After correcting for the lower detection limit, two Zn samples containing less than 0.1% Zn and one Pb sample containing more than 0.1% Pb had a difference of more than 15% between the Acme and ALS assay results. Overall, the results are acceptable and serve to confirm the results of the wider body of Acme lab samples.

11.3.4 Author's Opinion

The author, Mr. Jason Baker, considers the procedures used for the 2011 samples to be adequate for the purposes of this report.

12.0 DATA VERIFICATION

12.1 Gays River Deposit

As stated in Roy and Carew (2011), the sampling results were reviewed, and it was verified that the sample types and density were adequate for establishing Resources and Reserves. The sampling results are representative of the mineralization. The available information and sample density allow a reliable estimate to be made of the size, tonnage, and grade of the mineralization in accordance with the level of confidence established by the Mineral Resource categories in the CIM Standards.

12.1.1 Database Validation

A sample of 59 drill holes (4.3%) was selected for database validation. The collar locations, downhole survey data, geological logs, and assay data in the database was compared against the original, written logs.

12.1.1.1 Methodology

ScoZinc provided scanned original drill logs in Adobe (.pdf) format. An up-to-date copy of the electronic database of all drill hole information was also provided. An additional data file of drill hole coordinates was supplied, as many of the original drill logs did not have coordinates.

A total of 59 holes were selected (see Table 12-1). Most of the holes were located within areas with the highest economic potential, but the selection process also strived to provide good coverage for the whole deposit. This amounted to 4.3% of the more than 1,400 holes drilled on the property.

Printouts were made of the relevant sections of each of the holes, and of the assay data of the corresponding assay intervals. The assays were printed on the reverse of the drill logs. Coordinates on the log and database were manually compared.

12.1.1.2 Results

The data in the Excel database and original drill logs were manually analyzed. They were found to be, for the most part, comparable. Many of the original drill logs, both underground and surface, did not have collar coordinates or downhole survey data. Another database was located that contained the required information. It is more than likely that the holes were surveyed and the information was filed in a separate location from the original logs.

Table 12-1: Holes That Were Verified During the Database Validation

S61	S352	S613	S882	U047	U206
S69	S390	S634	S938	U057	U217
S71	S404	S648	S939	U061	U218
S85	S423	S663	S943	U073	U246
S94	S431	S690	S956	U087	U259
S110	S466	S703	S975	U093	U290
S183	S473	S705	S976	U106	U297
S220	S555	S726	S980	U129	U321
S251	S568	S843	U003	U148	U337
S268	S574	S857	U008	U174	

The following holes were found to have discrepancies between the original data from the drill logs and the final database.

S 69 Database 73.76-75.59 Pb 0.01% Original Log 73.76-75.59 Pb 0.32%.

S110 Assay data for database match that on original log. However, a hand-written correction on the log shows reduced Zn and Pb values.

S 663 Minor sample depth errors - not significant.

S703 Assays on original log for interval 89.0-99.83 m not shown. These were likely assayed at a later date.

S 726 Assay section on original log 77.72- 83.82 m (6.1 m) used on database. Original log interval was corrected by hand at a later date to 2 ft (0.61 m).

U 129 Sample from 115-125 ft (10 ft) misread as 115-128 ft (13 ft). Written entry on original log looks like 128 ft.

U218 Azimuth on database shows 235°, which is consistent with other angle holes with the same coordinates. However, a listing in another database shows an azimuth of 180°. It is more than likely that the database listing is correct.

12.1.1.3 Conclusion

With the exception of Hole S 110 and S 726, where significant assay intervals and values were involved, the remainder of the holes do not represent any factor that would change the status of the deposit. In general, the data transfer from the original logs was of high quality and the database was considered a valid representation of the mineral deposit.

12.1.2 Verification Sampling

The Scotia Mine property was visited by Mr. Reg Comeau of ACA Howe on 17 June, 21 June, 22 September, and 26 September 2004, in order to become familiar with the area and to conduct verification sampling on the property. Split, random, core samples were inspected and sampled from the site on the second visit during the 2004 drilling campaign. These core samples were in the area of the proposed low grade open pit, in the central portion of the deposit, as well as the Northeast zones' higher-grade area. A second set of core samples from the 1997 drilling campaign was later collected by Mr. Doug Roy.

Samples from the 1997 and 2004 drilling campaigns were collected, packaged, and independently shipped by Reg Comeau. All samples were taken from the remaining half core samples in the core boxes and were sawed in half, reflecting a quarter core sample. The remaining quarter core was left in the core tray. The samples were packaged and shipped to ACA Howe's office in Toronto, then subsequently shipped to and analyzed by SGS Toronto. The comparison of assay results is shown in Table 12-2.

The comparison of analytical results between the original 1997 SGS samples and the samples from the 2004 drilling program (analyzed at Minerals Engineering Centre of Dalhousie University) was excellent.

The author is satisfied that the assay data base for the property is sound and sufficient for the purpose of estimating resources and reserves.

Table 12-2: Results of Verification Sampling

				Original Assay		Howe Sampling	
Hole No.	**From (m)**	**To (m)**	**Interval (m)**	**% Zn**	**% Pb**	**% Zn**	**% Pb**
2004 Drilling Program by ScoZinc - Pit Area							
S968	2.70	4.70	2.00	3.38	0.29	3.62	0.14
S969	8.00	10.00	2.00	2.15	0.00	2.22	0.00
S971	2.90	4.90	2.00	4.63	0.00	3.91	0.00
S972	14.30	16.30	2.00	1.86	0.18	2.06	0.17
S973	74.00	75.00	1.00	11.90	14.98	14.18	17.25
S974	66.80	68.00	2.00	2.46	2.22	2.59	1.95
S976	98.10	98.45	0.35	7.66	0.23	7.19	0.17
2004 Drilling Program by ScoZinc - Northeast Zone							
S977	96.00	96.40	0.40	6.77	0.01	9.47	0.01
S982	133.30	133.60	0.30	0.84	0.32	0.84	0.18

				Original Assay		Howe Sampling	
Hole No.	**From (m)**	**To (m)**	**Interval (m)**	**% Zn**	**% Pb**	**% Zn**	**% Pb**
1997 Drilling Program by Westminer - Pit Area							
S926	18.40	19.90	1.50	2.82	0.01	3.16	<0.01
	19.90	21.40	1.50	3.27	0.01	2.86	<0.01
S933	12.10	13.60	1.50	1.40	0.01	1.47	0.01
	13.60	14.90	1.30	2.78	0.01	2.45	<0.01

S936	8.50	9.80	1.30	3.73	0.01	4.20	<0.01
	11.00	12.20	1.20	1.02	0.01	0.98	<0.01
1997 Drilling Program by Westminer - Northeast Zone							
S943	60.75	62.00	1.25	7.56	2.63	6.95	2.76
	62.00	63.00	1.00	3.16	5.70	2.78	3.30
S950	36.00	37.15	1.15	5.20	3.02	3.99	2.19
	37.15	38.25	1.10	17.37	1.07	15.54	0.67
S953	91.80	92.65	0.85	4.41	7.34	3.97	7.47

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

As discussed in Section 8.0, the lower-grade Zn / Pb deposit is of the MVT. The projected LOM mill feed grades were calculated to be 1.57% Pb and 3.06% Zn. During the 2008 operating period the average mill grades were 1.03% Pb and 2.70% Zn.

The projected metallurgical performance provides for a Pb concentrate grading 71% Pb at 85.7% recovery, and a Zn concentrate grading 57% Zn at 86% recovery. The projected performance has been achieved in testwork and historically and upside is not considered within the financial model. Improvements to debottleneck the process and reduce downtime are expected to improve plant availability, which will have a positive impact on grade and recovery. In addition, the planned plant automation will improve stability and reliability of operations, further supporting consistent performance and potential improvements. The benefits of the process improvements and automation have not been considered in the financial model, and remain to be quantified. The projected metallurgical performance is predicated on having competent operating and maintenance crews in place, a fully-functional assay laboratory, and an effective preventive maintenance program.

No deleterious minor elements are contained in the concentrates. The products should be readily marketable, given the clean high-grade nature of the concentrates.

13.2 Metallurgical Testwork

Selwyn completed a program of metallurgical testwork that evaluated a single composite sample of Zn / Pb mineralization from the Gays River project. The testwork was completed at ALS Metallurgy of Kamloops, British Columbia, and the results are contained in their report entitled “Metallurgical Testing of ScoZinc Mineralization – KM3677”. No other metallurgical test reports are available or have been reviewed.

The testwork completed at ALS Metallurgy indicated that high recoveries of both Zn and Pb can be expected from the material with high quality concentrates. A summary of the metallurgical performance of the ScoZinc mineralization, extracted from the ALS metallurgical report, is shown in Table 13-1.

Table 13-1: Metallurgical Test Results

Product	Mass	Assay - percent			Distribution - percent		
	%	Pb	Zn	Fe	Pb	Zn	Fe
Test 9 - Cycles IV and V							
Flotation Feed	100	1.83	2.46	0.9	100	100	100
Lead 2nd Clnr Con	2.4	71.7	3.00	0.5	93.5	2.9	1.4
Zn 3rd Cleaner Con	3.4	0.78	61.3	0.5	1.5	86.0	1.7
Zn 1st Cleaner Tail	3.7	0.76	2.75	1.5	1.5	4.1	5.9
Zn Rougher Tail	90.5	0.07	0.19	0.9	3.5	7.0	91.0
Test 10 - Cycles IV and V							
Flotation Feed	100	1.82	2.44	1.0	100	100	100
Lead 2nd Clnr Con	2.5	68.9	3.71	0.6	94.3	3.8	1.6
Zn 3rd Cleaner Con	3.6	0.72	59.0	0.5	1.4	86.9	1.8
Zn 1st Cleaner Tail	4.7	0.40	1.82	1.3	1.1	3.5	6.6
Zn Rougher Tail	89.2	0.06	0.16	1.0	3.2	5.8	90.0

Within the metallurgical test program at ALS Metallurgy, preliminary liberation analysis of the Zn / Pb mineralization was completed confirming the selection of particle size distribution for the restart of the operation.

The flowsheet used in the completion of laboratory testwork is significantly simplified from that currently in place at the ScoZinc site and modifications are recommended to simplify the process flowsheet.

13.3 2008 Plant Metallurgical Performance

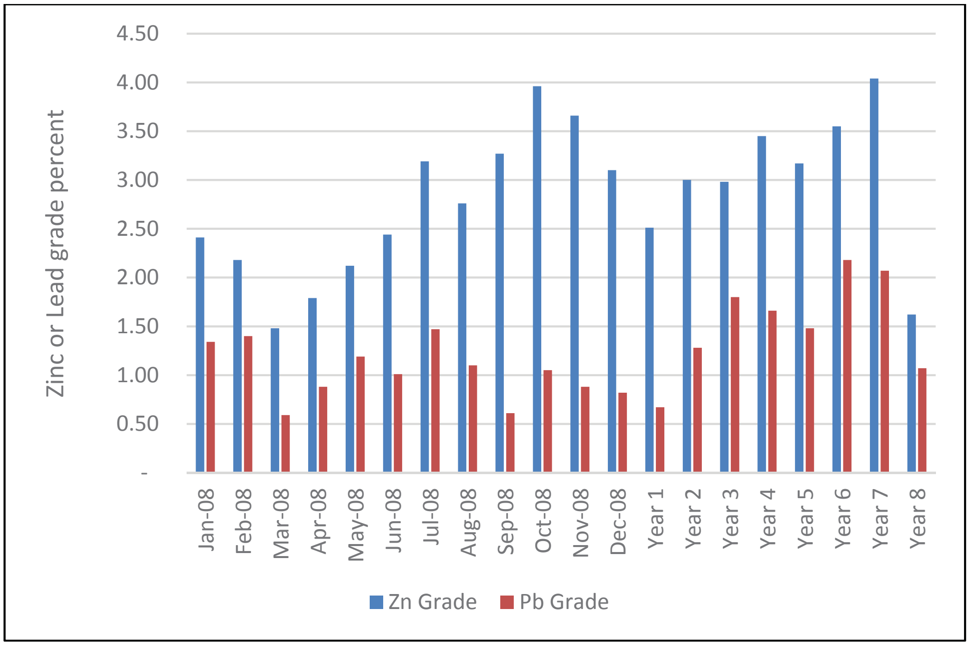
13.3.1 Mill Feed Grades

Figure 13-1 indicates the monthly feed grades that were reported from January to December in 2008. The Pb feed grades were erratic during the period, while the Zn feed grade increased each month, following the end of the first quarter.

The average projected feed grades for years 1 through 8 are shown, based on the mine plan values. The projected average Pb grades are higher those achieved in 2008, while the projected Zn grades are similar to those experienced during the final two months of operations in 2008. The higher Pb grade can be attributed to the higher Pb values within the resource model below the 460 m elevation, which is now included in the LOM plan. An increase in feed grades will generally enhance metallurgical performance. However, it may be necessary to adjust the mining schedule and/or implement ore stockpile management practices to minimize potential short term fluctuations in plant feed grade and maximize mill performance.

Expected Pb feed grades are generally below the grade of the test sample used at ALS Metallurgy, and Zn feed grades for the first 7 years of operation are higher than the grade of the test sample.

Figure 13-1: Comparison of 2008 Monthly and Mine Plan Annual Feed Grades



A comparison was also made with the daily mill feed grades from the 2008 operating period with the projected annual mill feed grades and the LOM average. The data presented in Figure 13-2 and Figure 13-3 demonstrate that the mill has operated within the range of grades expected during historical operations.

Figure 13-2: Daily Mill Zn Feed Grade 2008 Operations

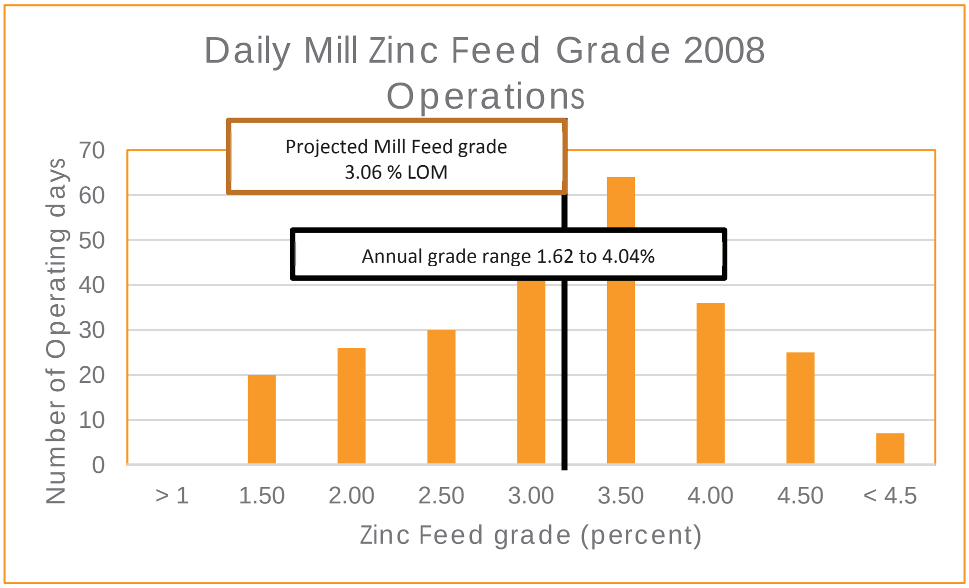
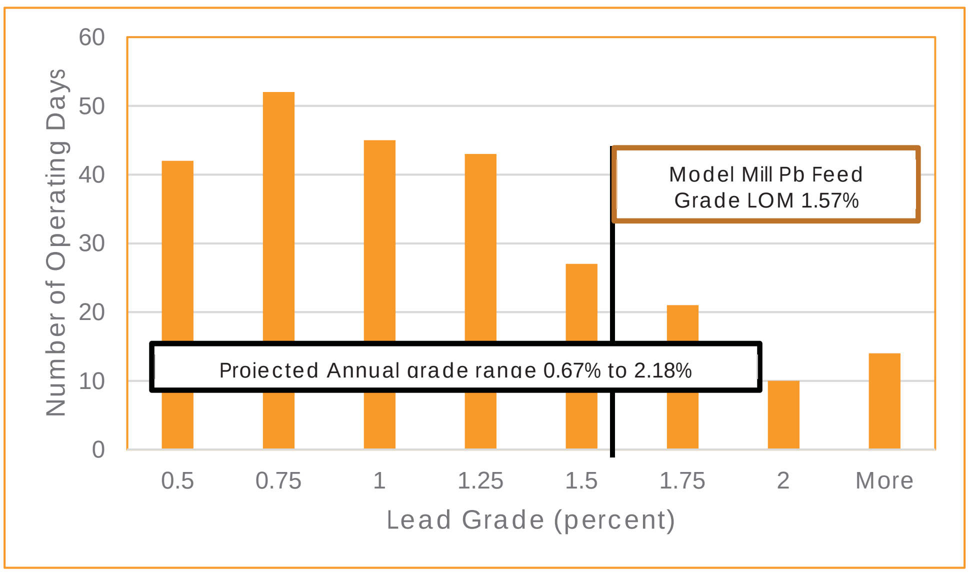


Figure 13-3: Daily Mill Pb Feed Grade Operations 2008



13.3.2 Metallurgy

The reported concentrate grades and metal recoveries reflect the relatively simple mineralogical characteristics of this type of mineralization. The mill process is designed to remove Pb in the primary recovery steps with the resultant tail to be processed for Zn recovery. The degree of variability experienced during 2008 was attributed to variation on feed grades, as well as plant bottlenecks and reliability issues. The restart capital plan has included expenditures to address the known deficiencies.

The ScoZinc operations were disrupted to varying degrees by mechanical problems and materials handling difficulties. The proposed changes are intended to correct the known mill deficiencies prior to the resumption of operations. The overall flowsheet is simplified to reduce circulating loads within the flotation plant. It is reasonable to assume that, given more stable operating conditions, the metallurgical performance will achieve a Pb concentrate grading 71% Pb at 85.7% recovery and a Zn concentrate grading 57% Zn at 86% recovery. Operational efficiency generally also results in reduction in mill operating costs.

The plant metallurgical results were influenced by fluctuations in feed grade: metallurgical results improved with increasing feed grades, and conversely deteriorated as feed grades decreased. This was particularly evident in the case of Zn metallurgical response. Metallurgical projections are conservative with no adjustment based on feed grade variation. The planned introduction of a primary Jaw crusher and stockpiles will allow for a blending program which may help alleviate grade variation and associated loss in mill efficiency.

The monthly Zn and Pb grade / recovery data for January 2008 to December 2008 are shown in Figure 13-4 and Figure 13-5, respectively, as provided in the monthly reports filed on SEDAR. In addition, financial model grade 57% and recovery 86% are shown as a point of reference. As can be seen in this data, there is no significant trend that can be attributed to normal metallurgical limitations. Rather, it appears that operational instability, possibly due to feed grade fluctuations, lack of plant control, or mechanical disruptions, are modulating the grade / recovery data.

Figure 13-4: Historical 2008 and Financial Model Zn Grade Recovery Data

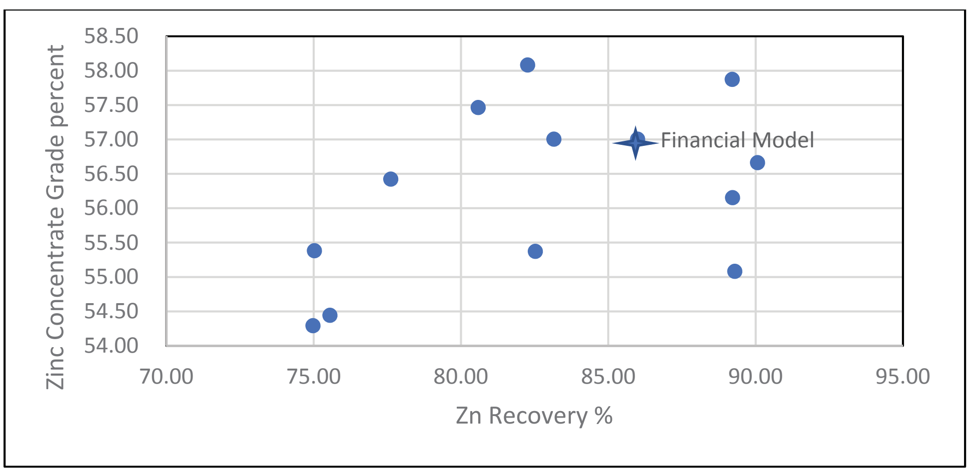
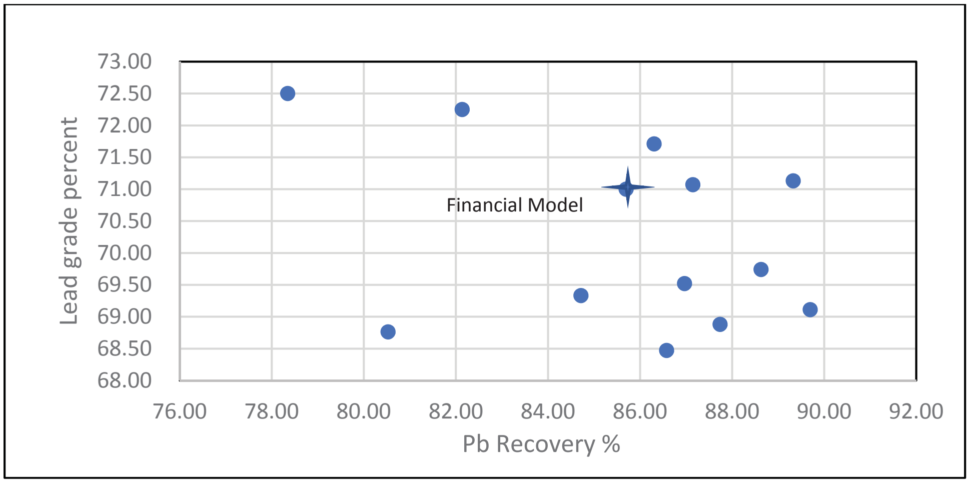


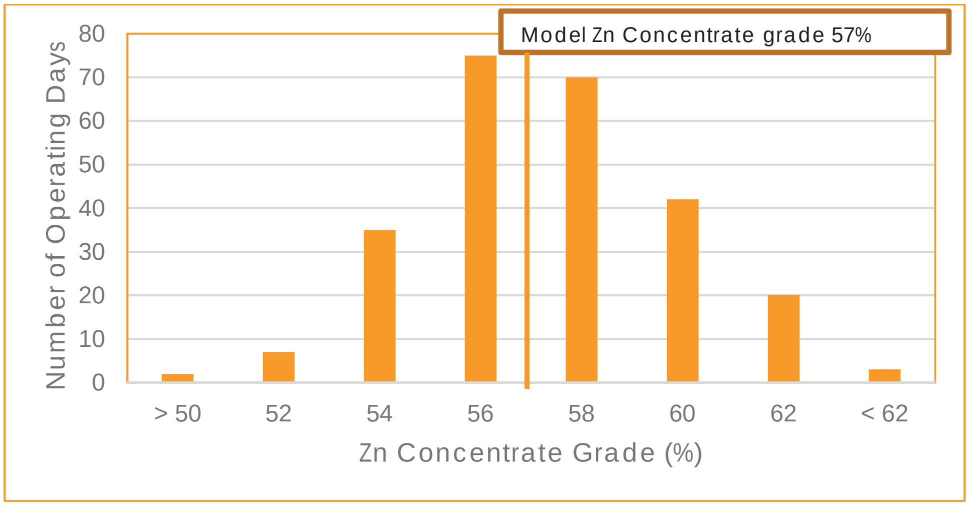
Figure 13-5: Historical 2008 and Financial Pb Grade Recovery Data



An examination of daily mill production data was also completed to determine if the assumed mill grade and recovery performance is reasonable based on the results achieved during the 2008 operating period. In addition, the distribution of the data could lead to some insight on the plant stability, variation to be expected, and identify parameters that are important to plant performance.

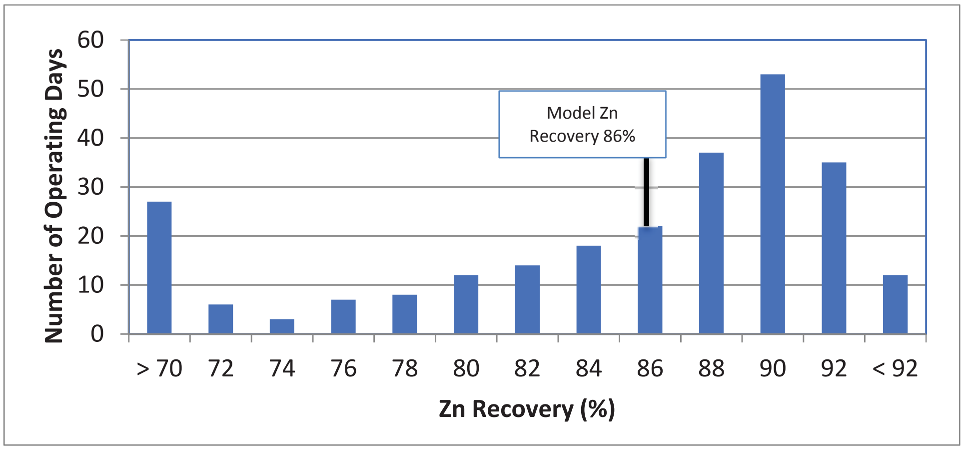
The Zn concentrate grade is projected to be 57%. A histogram of the daily results (presented in Figure 13-6) demonstrates that the assumption is appropriate and supported by the operating data. With the planned improvements and improved operating performance, the financial model Zn grade is expected to be consistently achieved.

Figure 13-6: Zn Concentrate Grade 2008 Operations



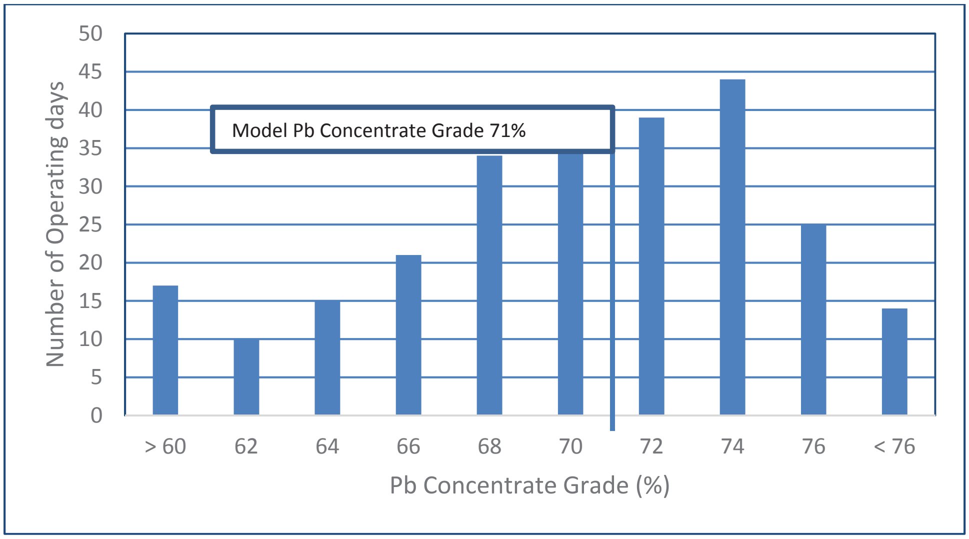
The Zn concentrate recovery is projected to be 86%. During the 2008 operating period, a histogram of daily results indicates that the projected recovery is appropriate and supported by the operating data. The spread on the operating results, shown in Figure 13-7, suggests that very poor operating performance with a recovery of less than 70% is a concern and improvements are required to ensure this is addressed in the restart capital improvement plan. The histogram in Figure 13-7 also has a significant distribution of data in the 88%–92% recovery, which suggests the at the selected 86% recovery value may be conservative if the poor performance days can be minimized. The capital improvements to address this are presented in Section 22.0.

Figure 13-7: Zn Concentrate Zn Recovery 2008 Operations



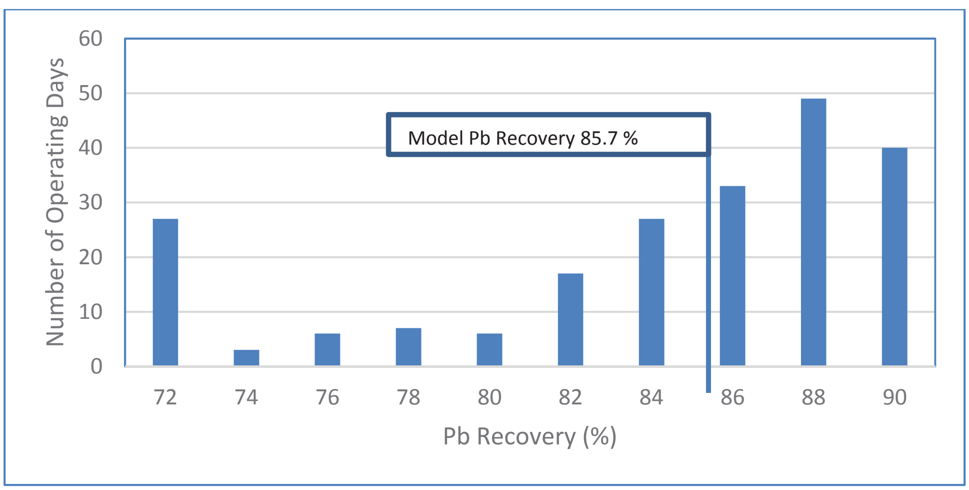
The Pb Concentrate Pb grade is projected to be 71%. A histogram of daily operating results from 2008 is presented in Figure 13-8 and suggests that the projected performance is appropriate. It is worth mentioning that the mill Pb feed grade is expected to be higher than what was experienced in the 2008 data as a result of the mineral distribution in the resource model. A significant spread in the data in Figure 13-8 was not attributable to any single factor, but suggests that improved plant control would help plant performance. Historic observations from operators indicated that poor grinding performance, sanding of Pb rougher cells, and lack of control of Pb flotation were prime causes of this distribution. These aspects are addressed in the re-start capital plan in Section 22.0.

Figure 13-8: PB Concentrate Pb Grade Operations 2018



The Pb Concentrate Pb recovery is projected to be 85.7%. A histogram of daily operating results from 2008 is presented in Figure 13-9 and suggests that the projected performance is appropriate. It is worth mentioning that the mill Pb feed grade is expected to be higher than what was experienced in the 2008 data. This is because the mineral distribution in the resource model indicated a higher recovery under normal circumstances. Similar to the Zn recovery data, a number of very poor operating days in Figure 13-9 was not attributable to any single factor, but suggests that improved plant control would help plant performance. Historic observations from operators indicated that poor grinding performance, sanding of Pb rougher cells, and lack of control of Pb flotation were prime causes of this distribution. These aspects are addressed in the restart capital plan in Section 22.0.

Figure 13-9: Pb Concentrate Pb Recovery 2008 Operations



No prior data is available with respect to the sensitivity of metallurgical performance to changes in flotation feed grind size reagent distribution and conditioning times or flotation circuit retention times. These parameters will change as a result of the increased mill feed rates proposed, and the higher feed grades.

Once operations are commissioned and stabilized, it is suggested that more comprehensive circuit sampling and mineralogical analysis be conducted to better understand relationships, to guide plant improvements, or to stabilize performance. Expected metallurgical recoveries, feed grades, etc., by year are shown in Table 17-1. Constant concentrate grades and recoveries are assumed throughout the LOM for Zn and Pb.

13.3.3 Metallurgical Accounting

Metallurgical projections are based on data provided in the ScoZinc daily / monthly reports from the 2008 operating period, as well as laboratory testwork. While there is no reason to doubt the validity of these data, a lack of relevant procedural documentation and reports precluded audits and validation of important metallurgical accounting information.

One of the key operational problems in past operating periods was the loss of Pb mineralization into the Zn circuit. This problem appears to be the result of not knowing the grade of Pb circuit tailings (feed to the Zn circuit) on a real time basis and inconsistent grinding circuit performance leading to liberation issues. This issue is expected to be corrected with the installation of an on-stream analyzer.

An on-stream analyzer is proposed in the mill circuit to allow the measurement of the following process streams for Zn, Pb, iron (Fe), and insoluble material:

- Flotation Feed
- Pb Rougher Tailings
- Final Pb Concentrate
- Final Zn Rougher Tailings
- Final Zn Concentrate
- First Zn Cleaner Tailings

Metallurgical accounting systems within the mill will be restored as part of the restart capital plan. There is no reason to doubt that the existing systems are suitable, as they were previously used for both metal accounting and concentrate management.

14.0 MINERAL RESOURCES ESTIMATE

The Gays River Deposit's mineral resource estimate was prepared by Douglas Roy, P.Eng. of MineTech International Limited, and Mr. Tim Carew, P.Geo. of Reserva International LLC. Only Mineral Resources were identified. No economics work, such as estimating capital and operating costs, that would be required for identifying Mineral Reserves was carried out. In other words, no Mineral Reserves were identified.

14.1 Gays River Deposit

Main Zone mineral resources, located south and west of Gays River, were estimated by Tim Carew, M.Sc., P.Geo., who was a co-author of the updated Mineral Resource report filed 08 October 2012 and is a Qualified Person under Section 1.1 of NI 43-101. Estimation of Main Zone mineral resources is discussed in Section 14.1.3.

Northeast Zone mineral resources, located underneath and northeast of Gays River, were calculated by Douglas Roy, M.A.Sc., P.Eng., who was the above-mentioned report's Principal Author and is a Qualified Person under Section 1.1 of NI 43-101.

The Main Zone mineral resources (discussed in Section 14.1.3) were originally modelled by Tim Carew for Savage Resources during 1998. Mr. Carew updated the model using linear unfolding for an NI 43-101-compliant resource estimated in 2006 (Roy et al., 2006). An update of the resource estimate was completed in 2011 (Roy et al., 2011). As there had been no new drilling in this zone since 2006, the significant changes from 2006 were:

- A re-tabulation of Main Zone mineral resources using the revised Zn-equivalent grade (for Pb – refer to Section 14.1.1); and
- Subtraction of the material that was mined during 2007 and 2008.

The current update of the resource estimate incorporates new drilling by Selwyn Resources in 2011, and a re-interpretation of the Main Zone model, based on a low-grade threshold of 0.5% Zn-equivalent, as opposed to the 2% threshold used in previous modeling.

Mr. Roy estimated the Northeast Zone's mineral resources in 2006 using a cross-sectional end-area method (Roy et al., 2006). For the current estimate, Mr. Roy re-estimated those resources using block modelling and carried out grade estimation using inverse distance weighting (refer to Section 14.1.3.9)

Though mineral resources for the Main and Northeast Zones were estimated separately, they abut one another and represent a single, geologically continuous, mineralized body.

14.1.1 Zinc-Equivalent Grade

For cut-off grade purposes, Pb's Zn-equivalent grade was calculated and added to the Zn grade (see

Equation 1).

The Zn-equivalent grades were calculated using metal prices derived from 2011 (the time of the previous report) and going-forward LME contract prices, recovery values from previous mill production, and typical smelter return values (1% Pb is equivalent to 1.2% Zn). For simplicity, Stantec used these assumptions and methodology for the basis of this report. The Zn plus Zn-equivalent grade was added to the block model field "Zn-Eq".

Equation 1: Pb's Zn-Equivalent Grade, Using Historical Data

$$1\%\ \text{Lead} = \frac{\text{Lead Price}}{\text{Zinc Price}} \times \frac{\text{Lead Recovery}}{\text{Zinc Recovery}} \times \frac{\text{Lead Smelter Return}}{\text{Zinc Smelter Return}}$$

$$= \frac{\$1.05}{\$1.00} \times \frac{86\%}{84\%} \times \frac{95\%}{85\%}$$

$$= 1.20\%\ \text{Zinc}$$

1. Recovery values are actual values from 2008.
2. Smelter returns were estimated.
3. Metal prices were supplied by Selwyn on Aug 7, 2012.

14.1.2 Specific Gravity / Density

Prior to 2007-2008, there was no record of any systematic whole-rock SG measurements being taken. Therefore, in the 2011 report, a formula for specific gravity based on Zn and Pb grades was used for the mineralized zones. This formula, which was also used by Savage Resources for their 1998 resource estimate, was:

SG = 1 / (Pb%/ (86.6 × 7.6) + Zn% / (67.0 × 4.0) + (1 – Pb% / 86.6 – Zn% / 67.0) / 2.7)

Selwyn undertook SG measurements on core from the 2011 drilling program, with 559 determinations in all and 250 determinations on intervals above the mineralized threshold of 0.5% Zn-equivalent. On average the formula overestimated the SG by 0.4%, with a standard deviation of 3%. This difference is not considered to be material, and the formula-estimated values were retained for the 2011 estimate. Stantec will continue to use this methodology in this report.

14.1.3 Main Zone Resources

14.1.3.1 General

The deposit is characterized by complex geometry and is difficult to model in terms of standard techniques. Lying along a paleo-shoreline, it features repetitive changes in strike of 90° or more around a general trend of 060° Azimuth, with varying dip. This geometry makes it difficult to incorporate the true spatial relationship of the samples for estimation purposes without the use of unfolding techniques. Unfolding transforms the sample data into another coordinate space that honours the spatial relationships. Variography and estimation are conducted in the transformed space, and the results are then back-transformed into the original space. The deposit has been mined by underground methods in the past and is therefore intersected by numerous openings along the hanging wall contact.

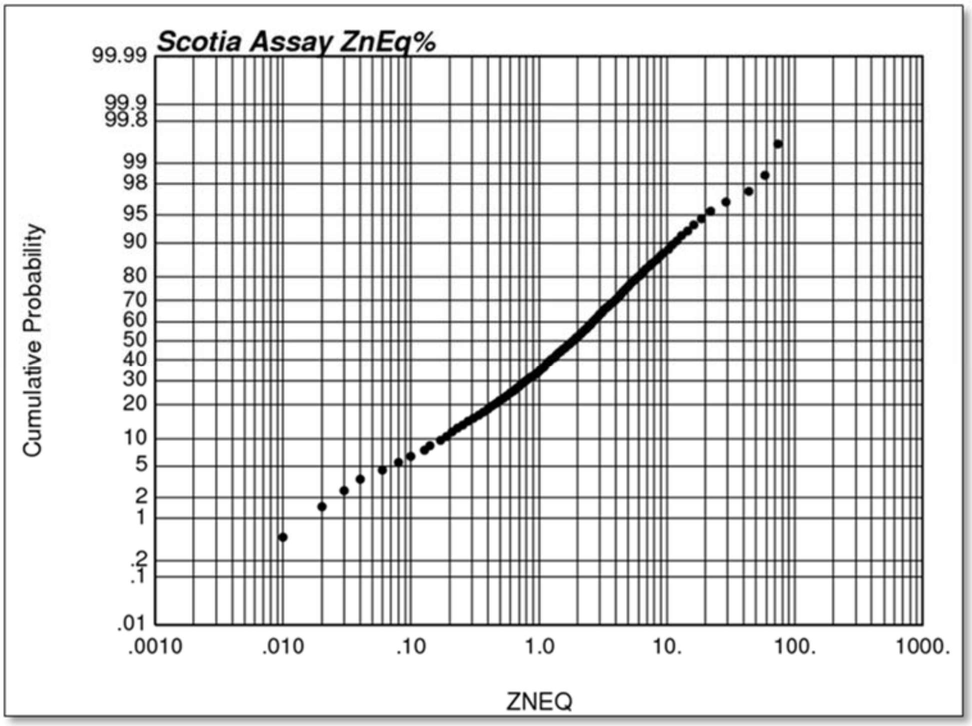
14.1.3.2 Geological Modelling Approach

Topographic contour data derived from the AutoCAD drawing files provided was utilized to create a triangulated surface (TIN) of the current topography over the project area, including open pit mining areas.

As determined in the original (1998) modeling, the geometric complexity and nature of the deposit requires manual interpretation, and that the ore zone be differentiated into a high-grade massive sulphide zone and a low-grade disseminated zone that occurs largely on the footwall side of the high-grade zone. For that modeling, a set of 3D solid models of the existing underground development and stope areas, developed by Mr. Bruce Hudgins of Hudgtec Consultants, was imported from AutoCAD DXF files provided. The drill hole data and underground openings were plotted on hard-copy plans at a 10-m interval, and interpretations of the high-grade zone, the low-grade zone, and the hanging-wall trench were produced. The cutoff grades that were used for the high-grade and low-grade zones were 7% Zn-equivalent and 2% Zn-equivalent, respectively. These values were selected to correspond with cutoffs utilized in earlier resource evaluations. The plan-view interpretations were digitized as closed polygons, then tied together in the GEMS solids modelling system to create separate 3D solid models of the high-grade, low-grade, and trench zones of the deposit. These models were adopted for use in the resource estimate of 1998 (Carew, 1998) and subsequent updates in 2006 (Roy et al., 2006) and 2011 (Roy et al., 2011).

The 2011 drill-hole data was added to the GEMS project files, and a new interpretation of the low-grade zone was produced using the revised low-grade cut-off of 0.5% Zn-equivalent. This threshold was selected with reference to the log-probability plot of assay Zn-equivalent values coded as Gays River Formation (carbonate), which exhibits a flexure point between low-grade mineralization and background mineralization at 0.5% Zn-equivalent (Figure 14-1).

Figure 14-1: Log-Probability Plot – Carbonate Zn-Equivalent Assays



Plan view contours of the existing 3D solids were used as a base for this purpose, with vertical sections cut on drill hole fans as required to refine the interpretation. An updated low-grade 3D solid was then generated from these plan interpretations for use in block modeling and resource reporting. In addition, a mineralized zone that was not modeled in earlier estimates was included in this update. Known as the Southwest Zone, it is a southerly continuation of the Main Zone mineralization that, although is adequately drilled in the newly modeled area, does not have sufficient drilling to define the paleo-shoreline geometry as expressed in the Main Zone.

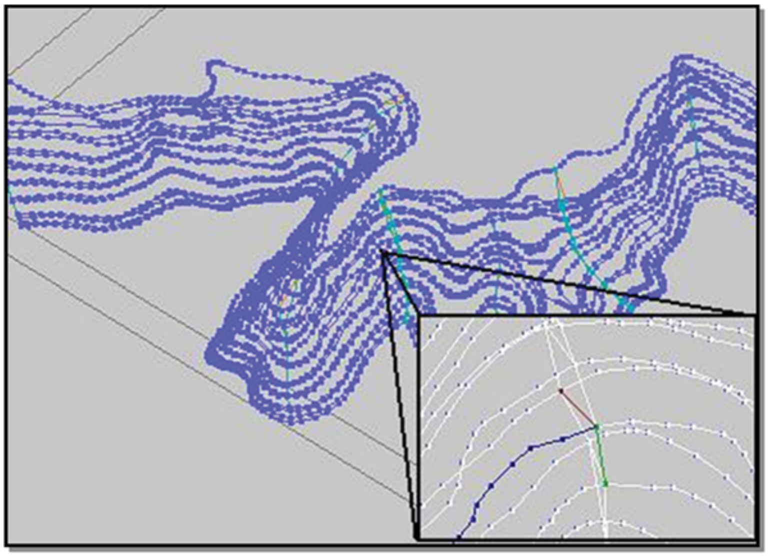
14.1.3.3 "Unfolding" Process

As stated in Section 14.1.3.1, the deposit is characterized by complex geometry and is difficult to model in terms of standard techniques. An unfolding technique was used that transformed the sample data into another coordinate space while honoring the spatial relationships.

The Gemcom GEMS unfold application was used for the transformation in this case. This approach is based on the concept of slabs – a slab being a region of space that is topologically equivalent to a cube. The edges are 3D polylines and are not necessarily straight from end to end. Each face is defined by four polylines on its perimeter and the nominally vertical edges of the slab may also have more than two points. The geological feature of interest, e.g., a folded and/or faulted vein or seam is broken down into a collection of adjacent slabs, the only proviso being that any two adjacent slabs must share an entire common face. The algorithm highlights three of the edge polylines of a representative slab that are nominally orthogonal and allows them to be associated with X, Y, and Z axes of the unfolded space. All the polylines are then categorized into three sets of lines corresponding to these unfolded axes. The unfolded slabs are displayed below the original polylines, and the unfolded lines will be aligned approximately to the X, Y, and Z axes. The average length of each of the sets is calculated and a nominal graticule size or spacing is entered. The unfolding transformation includes two graticules – one for the folded region and one for the unfolded region. The points in the two graticules have an exact 1:1 correspondence, which provides for a check that the transformation will be reasonable. If any graticule cells are highly skewed, for example, the folded region can be subdivided into smaller slabs. In addition, the interior vertices can be allowed to slide on the various sets of lines in order to minimize distortion.

The graticule points are simply samples of the transformation, and are connected by straight lines to make the visualization easier. Various combinations of the sliding axes can be experimented with, particularly in cases where the polyline lengths along the feature are different, in order to minimize the distortion in these cases. The 3D polylines were generated by contouring the 3D solid of the low-grade zone. These polylines were subdivided into a series of smaller adjacent slabs corresponding to the alternating strike direction of the deposit. A section showing the slabs and the allocation of the association with the unfolded axes is illustrated in Figure 14-2. The unfolded space is illustrated later in Figure 14-6.

Figure 14-2: 3D Polyline Slabs and Axes



The basic procedure is as follows.

- Creation of the unfolding transformation;
- Forward transformation (unfold) of the sample data points;
- Spatial analysis and block modelling in the transformed space;
- Back-transformation of the estimated block data (Zn and Pb) into normal (folded) space; and
- Allocation of the values to a block model in normal (folded) space by nearest neighbor interpolation.

14.1.3.4 Drill Hole Data

A subset of the overall drill hole database was utilized for estimation purposes, comprising those drill holes that intersected the 3D solid model of the carbonate mineralization. This subset comprises 662 holes, including the most recent 2011 drilling, and includes both surface and underground drilling.

14.1.3.5 Mineralized Envelope

The mineralized envelope for estimation purposes was restricted to the carbonate material within the 3D solid models created from plan view interpretations. These interpretations and 3D model are regarded as the most representative constraints on the mineralization available. Separate 3D models were developed for the low grade, disseminated portion of the deposit, and for the less continuous higher grade zone that lies along the footwall contact, and which was partly exploited by previous underground mining.

14.1.3.6 Statistical Analysis and Capping

The sample sets for Zn and Pb mineralization comprised those assay intervals falling within the 3D solids and were compiled separately for the lower-grade and higher-grade zones. The sample statistics, histograms and probability plots are shown in Figure 14-3 and Figure 14-4.

While both Zn and Pb assay grades exhibit fairly typical positively skewed distributions, the Pb values exhibit eviden ce of a multi-modal distribution, with a set of values falling in the 0.01 to 0.1% range. This may be related to the use of arbitrary and variable values for detection limits in the Pb data. In general, Zn and Pb values are not particularly well correlated, with a correlation coefficient of 0.32. There is also some evidence of possible misclassification of some values between low grade and higher grade zones in both cases, either in terms of original typing, or in geometric boundary effects relative to the 3D solids. The Zn values are generally well behaved, with relatively low coefficients of variation (COV), whereas the Pb values exhibit a relatively high COV.

Whereas initial studies on the deposit by Savage Resources Canada Co. considered a capping value of 13% on both Zn and Pb, examination of the probability plots indicates that although the number of high values steadily decreases, the upper tail for the all distributions are fairly continuous and unbroken up to values considerably higher than this, suggesting that higher capping values could be utilized. In later studies, discussions with Savage personnel led to an alternative approach in which the high-grade outliers in the distributions were retained in the data set prior to any compositing, but restricted in terms of interpolation. Block centroids were required to be within 5 m of the sample before it could be used in the estimation of the block in question. Given the indication that higher capping values could be considered, and to maintain consistency, this approach was adopted at that time and is retained for this study. No grade capping was applied prior to compositing, but the range restriction was subsequently applied in estimation for Zn and Pb composites above 20%.

Figure 14-3: Zn and Pb Assay Lognormal Histograms

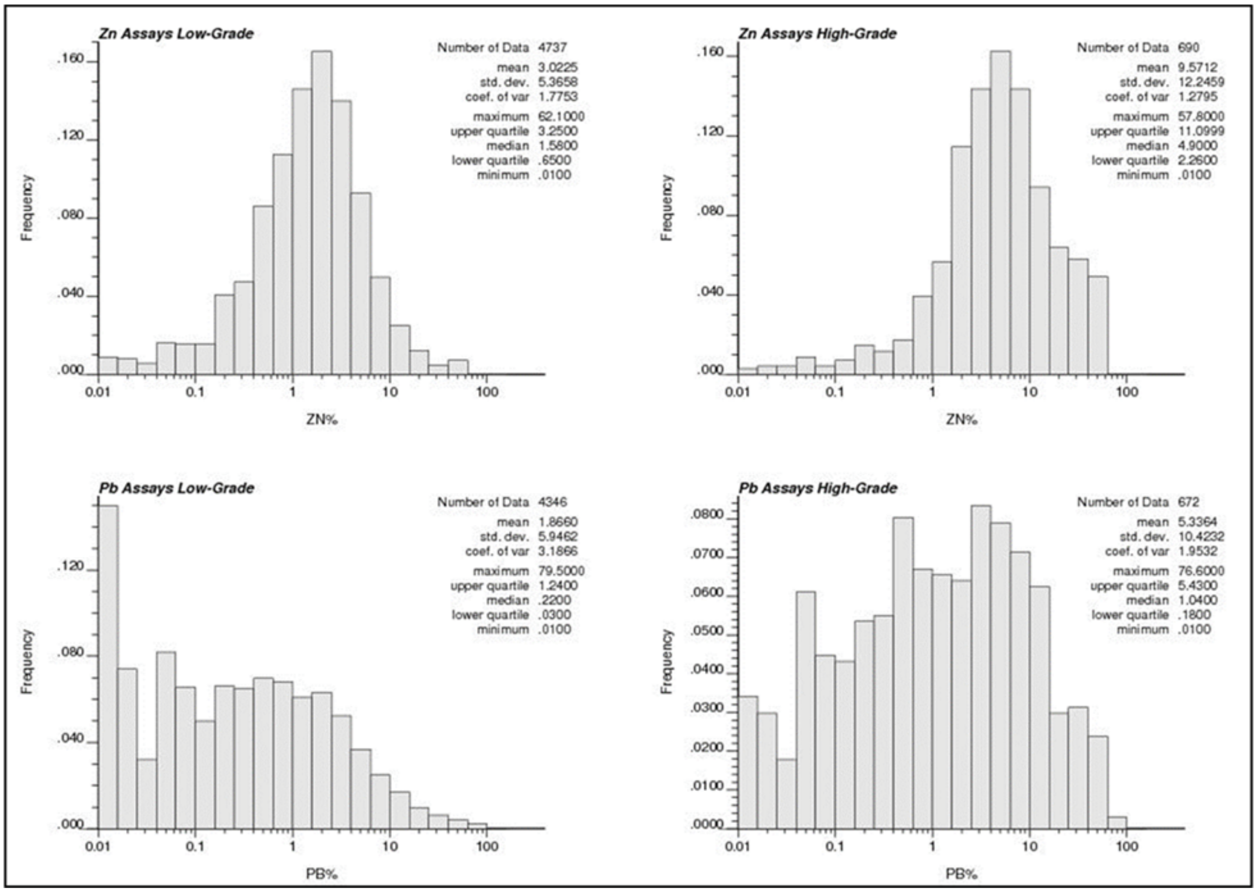
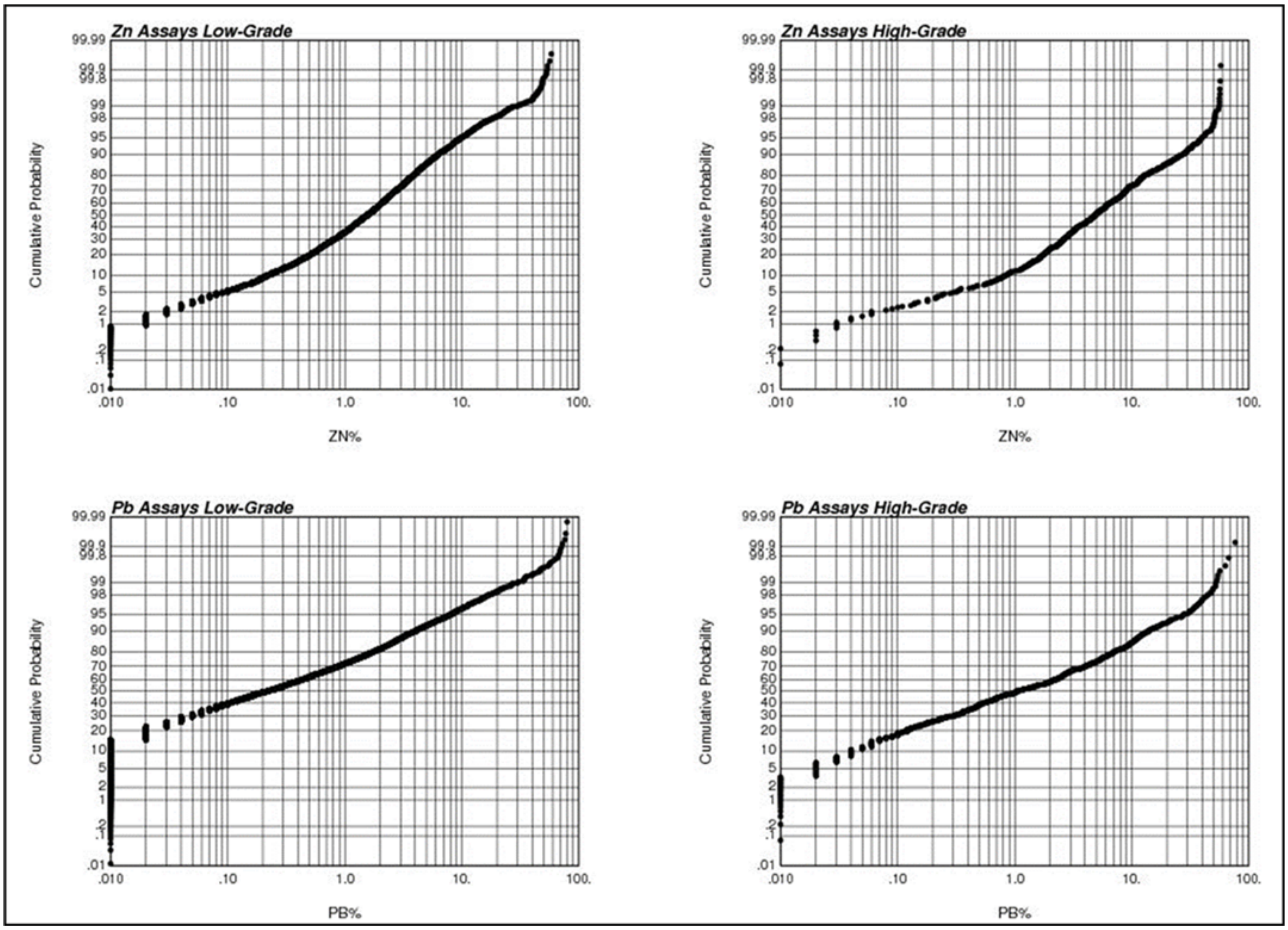


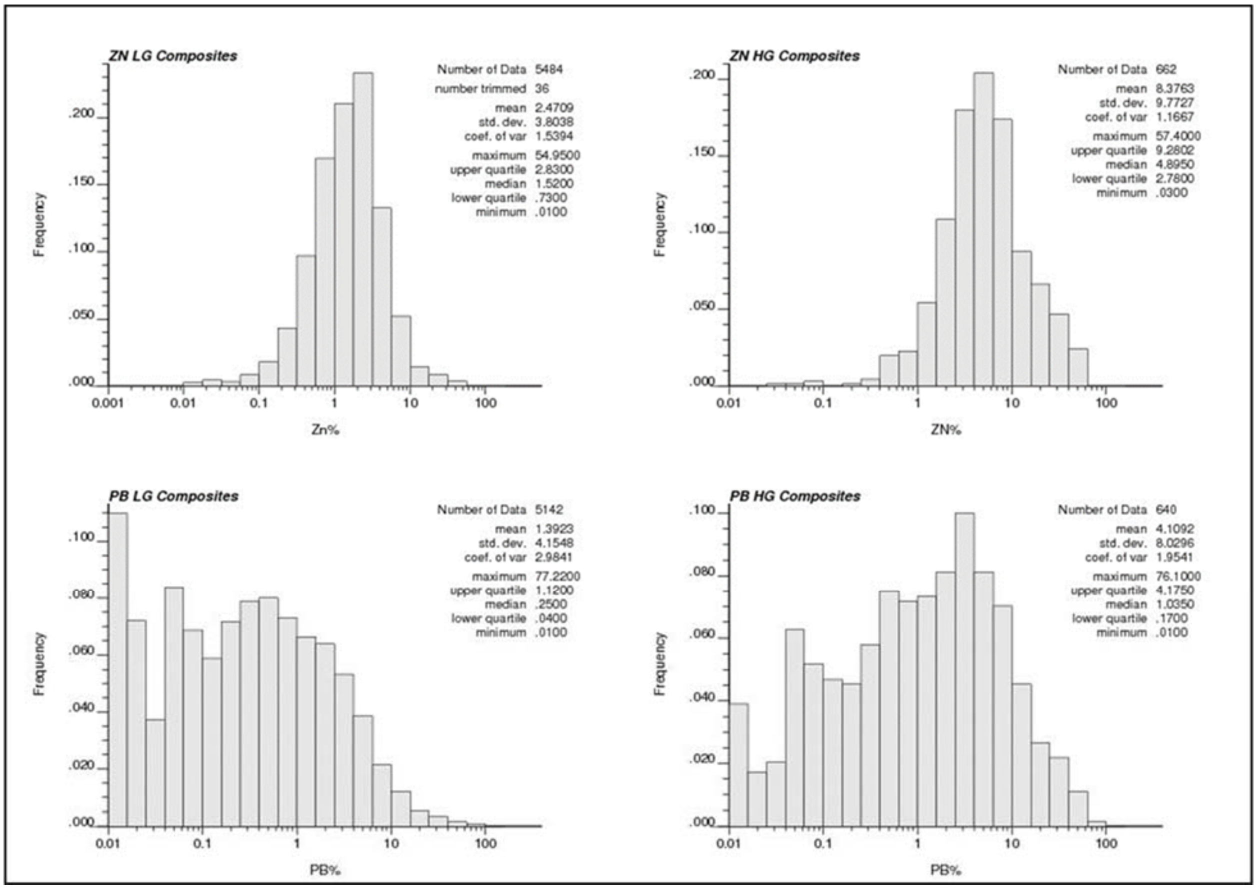
Figure 14-4: Zn and Pb Assay Probability Plots



14.1.3.7 Compositing

Equal length composites were prepared from uncut assay values in a two-step process. Initial composite intervals were defined from the intercepts of the drill holes with the high-grade and low-grade 3D solids of the mineralized zone. Equal length composites of 1.5 m were then generated within these intervals. The average length of the assay intervals is approximately 1.5 m. Residual intervals of less than 1.5 m at the top and bottom contacts were retained if the length was at least 0.6 m (40% of composite length). Intervals less than 0.6 m in length were discarded. The low-grade composites set was further subdivided into those falling below 490 m elevation (below which the deposit dips at varying angles) and those above 490 m, where the deposit is essentially flat-lying. The composite statistics and histograms for the overall higher grade and lower grade Zn and Pb are shown in Figure 14-5.

Figure 14-5: Composite Statistics and Histograms



14.1.3.8 Spatial Analysis

Three dimensional experimental correlograms were generated using the transformed (un-folded) Zn and Pb composite data, for both low-grade and high-grade mineralized zones below an elevation of 490 m. Separate 3D experimental correlograms were generated using un-transformed composite data for the low-grade mineralized zone above 490 m elevation, where the deposit is essentially horizontal in attitude. The resulting experimental correlograms are not considered robust enough for use in estimation by kriging, but did provide some indications with regard to suitable search distances and orientations to be used for estimation by Inverse Distance Squared (IDP2) interpolation.

14.1.3.9 Block Model and Grade Interpolation

Two block models were constructed for interpolation purposes, a primary model in normal (un-transformed) space, and a secondary, smaller model in transformed space for interpolation of the un-folded data. The primary block model was defined to cover the volume of interest, with the following Gemcom GEMS® parameters:

- Origin: 7200.00E / 6592.82N / 375.00 AMSL (Lower Left)
- Block Size: 5 m × 5 m × 5 m
- Columns: 160
- Rows: 400
- Levels: 45
- Rotation: -60° (To align with overall strike of deposit – Azimuth 060)

The primary block model is configured as a partial block model, which allows the percentage of various rock types within the block to be stored and utilized for manipulation and reporting purposes. The rock type model was initialized with the default rock code for air and all blocks below the current topographic surface were set to the Evaporites (gypsum) rock code. The model was then overprinted with rock codes for the overburden, trench, and Goldenville (quartzite) using 3D solids created from surfaces and sectional interpretations. This rock type model is referred to as the standard rock type model. The final step was overprinting with rock codes for the existing U/G mining excavations, the high-grade (HG) mineralized zone, the low-grade (LG) mineralized zone, and a solid created from the current topographic surface to represent material mined out in open pit mining in 2007-2008. The percentage of these four material types in blocks intersecting the solids was calculated and stored separately, with the mined-out solid having the highest priority, followed by the U/G excavations, high-grade zone and low-grade zone, in blocks where the solids overlapped. This procedure ensures that the mined-out material in the model is correctly accounted for. The rock code for any other material in these blocks was taken from the standard rock type model, i.e., a block on the hanging wall contact might comprise 50 % U/G excavation, 20 % HG zone, and 30 % trench material.

The 3D solid of the existing U/G excavations was generated by Mr. Bruce Hudgins of Hudgtec Consulting and was originally supplied by ScoZinc. The 3D solids of the HG and LG zones were generated from plan interpretations. Zn equivalent cutoffs of 0.5% and 7% were utilized for the LG and HG zones, respectively, in developing the interpretations.

Inverse distance squared (IDP2) interpolation was used to estimate Zn and Pb block values in the flat lying portions of the deposit above 490 m elevation. This estimation was restricted to the LG zone, as the HG zone does not extend above this elevation, and includes the South-West zone, which currently has no defined HG zone. The estimation was done in three passes with parameters as follows:

Pass 1

- Minimum # of samples: 3
- Maximum # samples: 8
- Max. # samples/hole: 2 (ensures that samples come from at least 2 holes)
- Search Radius/Direction: See Table 14-1 for details.

Table 14-1: Search Ellipse Details

	Ranges					
	Maximum		Intermediate		Minimum	
Zone	m	Az/Dip	m	Az/Dip	m	Az/Dip
LG>490	35	46/0	20	136/0	6	0/90

Pass 2

- Minimum # of samples: 3
- Maximum # samples: 8
- Max. # samples/hole: 2 (ensures that samples come from at least 2 holes)
- Search Radius/Direction: Pass 1 × 2

Pass 3

- Minimum # of samples: 3
- Maximum # samples: 8
- Max. # samples/hole: 0 (no restriction)
- Search Radius/Direction: Pass 2 × 2

Mineralized blocks in the dipping portion of the deposit below 490 m elevation were populated separately following interpolation in transformed space and back-transformation of the generated values (at block centroids) into normal space, as described below. The back-transformed data was then used to interpolate the Zn, Pb, and Classification values in normal space by the nearest-neighbour technique, separately for the LG and HG zones.

The secondary block model is a standard block model (every block has only one rock code), defined in 3D space to cover the volume of interest. As described earlier, the transformation process associates three of the edge polylines of a representative slab that are nominally orthogonal with the X, Y, and Z axes of the unfolded space. This space is orthogonal with respect to the original coordinate axes, and offset by a specified amount. The transformation selected in this case results in a space in which the X-axis corresponds to the unfolded strike component of the deposit (approximately 3,050 m), the Y-axis with cross-strike component (12 m), and the Z-axis with the down-dip component (143 m), as shown in Figure 14-6 and Figure 14-7, which also show transformed and un-transformed composite data.

Figure 14-6: 3D View - Transformation and Block Model Definition

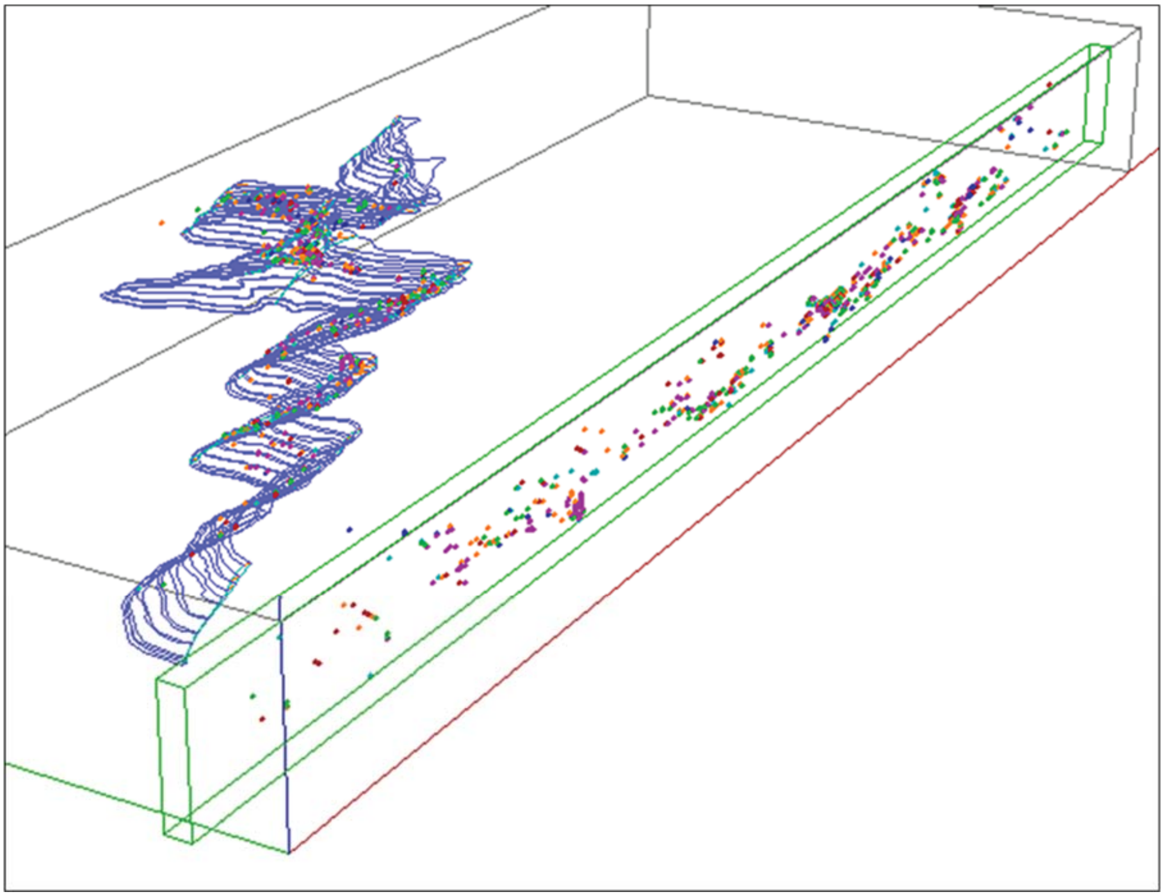
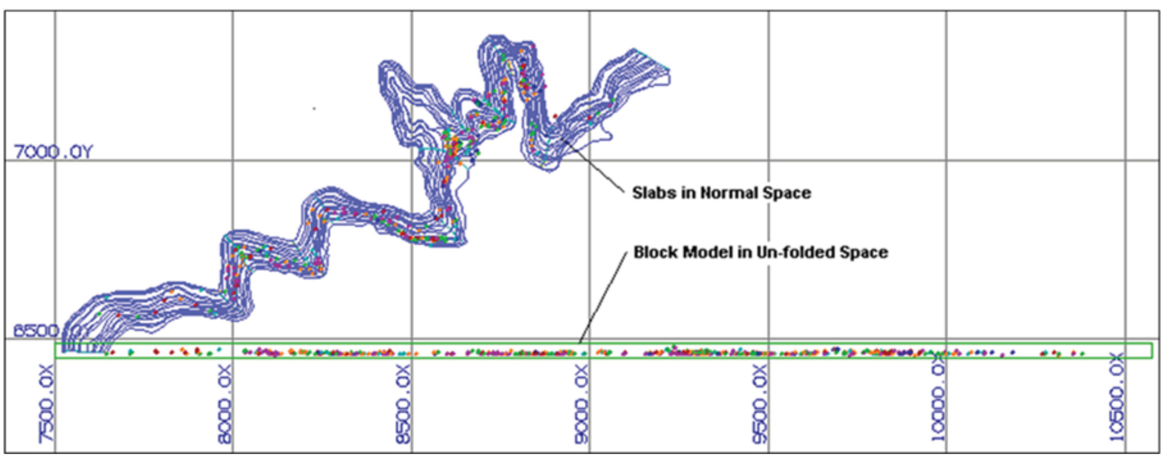


Figure 14-7: Transformation - Plan View



The secondary block model definition is as follows:

- Origin: 7500.00E / 6425.00N / 200.00 AMSL (Lower Left)
- Block Size: 7.5 m × 5 m × 5 m
- Columns: 410 (7.5 m)
- Rows: 15 (5 m)
- Levels: 40 (5 m)
- Rotation: No rotation

Separate interpolations of Zn and Pb block values for the LG and HG zones were estimated in three passes using Inverse Distance Squared (IDP2) interpolation and the transformed composites. The parameters were as follows:

Pass 1

- Minimum # of samples: 3
- Maximum # samples: 8
- Max. # samples/hole: 2 (ensures that samples come from at least 2 holes)
- Search Radius/Direction: See Table 14-2 for details.

Table 14-2: Search Ellipse Details

	Ranges – Transformed Model					
	Model East -X		Model North-Y		Model El.-Z	
Zone	m	Az/Dip	m	Az/Dip	m	Az/Dip
LG – Zn <490	30	90/0	7.5	0/0	15	0/90
HG - Zn	30	90/0	7.5	0/0	15	0/90
LG – Pb <490	30	90/0	7.5	0/0	15	0/90
HG – Pb	30	90/0	7.5	0/0	15	0/90

Pass 2

- Minimum # of samples: 3
- Maximum # samples: 8
- Max. # samples/hole: 2 (ensures that samples come from at least 2 holes)
- Search Radius/Direction: Pass 1 × 2

Pass 3

- Minimum # of samples: 3
- Maximum # samples: 8
- Max. # samples/hole: 0 (no restriction)

Search Radius/Direction: Pass 2 × 2

15.0 MINERAL RESERVE ESTIMATE

No mineral reserves have been established for this project since the economics of the project have not yet been demonstrated by a prefeasibility or feasibility study.

For the purposes of production scheduling for economic modelling in this PEA, a total open pit mineralized material and waste tonnage has been defined and is described in Section 16.1.3.

16.0 MINING METHODS

The resource will be mined using conventional truck and shovel mining methods in the open pits and Cut-and-Fill mining in the underground.

The open pits will mostly be mined on 10-meter-high benches by a mining contractor using the appropriate equipment to meet the material movement requirements laid out in the production schedule. This is a deviation from the 2011 PEA where an owner/operator approach was taken. Drilling and blasting will be required for the rock portion of the deposit, while the overlying overburden, which makes up approximately 60% of the waste to be mined, is considered free digging and will not require blasting. Drilling and blasting will also be done by a contractor.

The underground operation will be accessed from the lower benches of the open pit to reduce waste development costs and to use the open pit excavations and facilities for water management. Based on challenges faced in historic operations, the mine dewatering system (a combination of in-pit pumping and dewatering wells) has been sized to ensure sufficient capacity for the planned mining operations.

16.1 Open Pit Mines

16.1.1 Pit Optimization

Prior to designing the operating open pits in the 2013 PEA, a series of pit economic optimizations were run utilizing the Economic Planner application within MineSight software to define the optimal pit size and their configurations. The optimized pits were based on selecting the maximum NPV calculated by MineSight Economic Planner (no capital costs included) and logical mining paths. The optimized pits developed for the 2013 PEA were used for this study.

The inputs to the 2013 analysis were based on preliminary estimates and are as follows:

- Metal Prices: Lead = US$ 1.20/lb Zinc = US$ 1.10/lb
- Overburden mining cost: $2.20/t
- Waste rock mining cost: $2.20/t
- Ore mining cost: $3.07/t
- G&A cost: $6.94/t
- Rock pit slopes: 45°
- Overburden pit slopes: 22°
- Mining dilution: 7.5%
- Diluting grade: Lead = 0.50% Zinc = 1.00%
- Mining losses: 5%

16.1.2 Pit Design Criteria

In the 2013 PEA, the optimized pit shells were used as a design guide. Operating pits' shapes were created and used for mine scheduling. Allowance was made for incorporating truck access ramp designs into future detail pit plans at that time. Different slope angles were used in the weaker overburden and the harder rock. Table 16-1 summarizes the 2011 pit design criteria which Stantec will also use in this study. Note that no pit wall geotechnical investigations were completed nor any pit slope geotechnical studies, but these slope angles are reasonable for this stage of study. The plan is to initiate geotechnical surveillance and monitoring as part of the pit dewatering scope, and continue studies in preparation for operations. The geotechnical scope will be further defined based on initial findings.

Table 16-1: Pit Design Criteria

Overburden	
Bench Height	5 m
Berm Width	4.5 m
Batter Face Angle	35º
Inter Ramp Angle	28º
Rock and Gypsum	
Bench Height (double benching)	10 m
Berm Width	4.50 m
Batter Angle	70º
Inter ramp angle	50º
Haul Road Width	20 m
Haul Road Gradient	10%

16.1.3 Pit Tonnages

Using the pit optimization shells from the 2011 PEA, Stantec created a more detailed pit design with ramps and schedule for the first 4 years of production, as seen in Table 16-1. The pit was scheduled by month for the first 2 years and quarterly for years 2–4. Years 5–8 correspond to phases 5b, 6, 7, and 8 from the 2013 PEA design, which Stantec has utilized in this study. For this study, Stantec has divided the main pit and the northeast pit into five phases, shown in Figure 16-2. The total open pit resource mined from both pits is 6.37M t at a produced grade of 3.00% Zn and 1.50% Pb.

Table 16-2 presents a summary of the resource and waste tonnages, as well as Zn and Pb, provided by each open pit phase. Waste has been subdivided into overburden, gypsum, and carbonate waste rock. The pit phase sequence is shown in Figure 16-2. A detailed layout of the month to month pit progression for the first 4 years of production can be found in Appendix 1. The bench-by bench, end-of-period, mining cuts showing the location of the historic underground workings are also shown in Appendix 1.

Figure 16-1: Detailed Pit Design YR1 to YR4

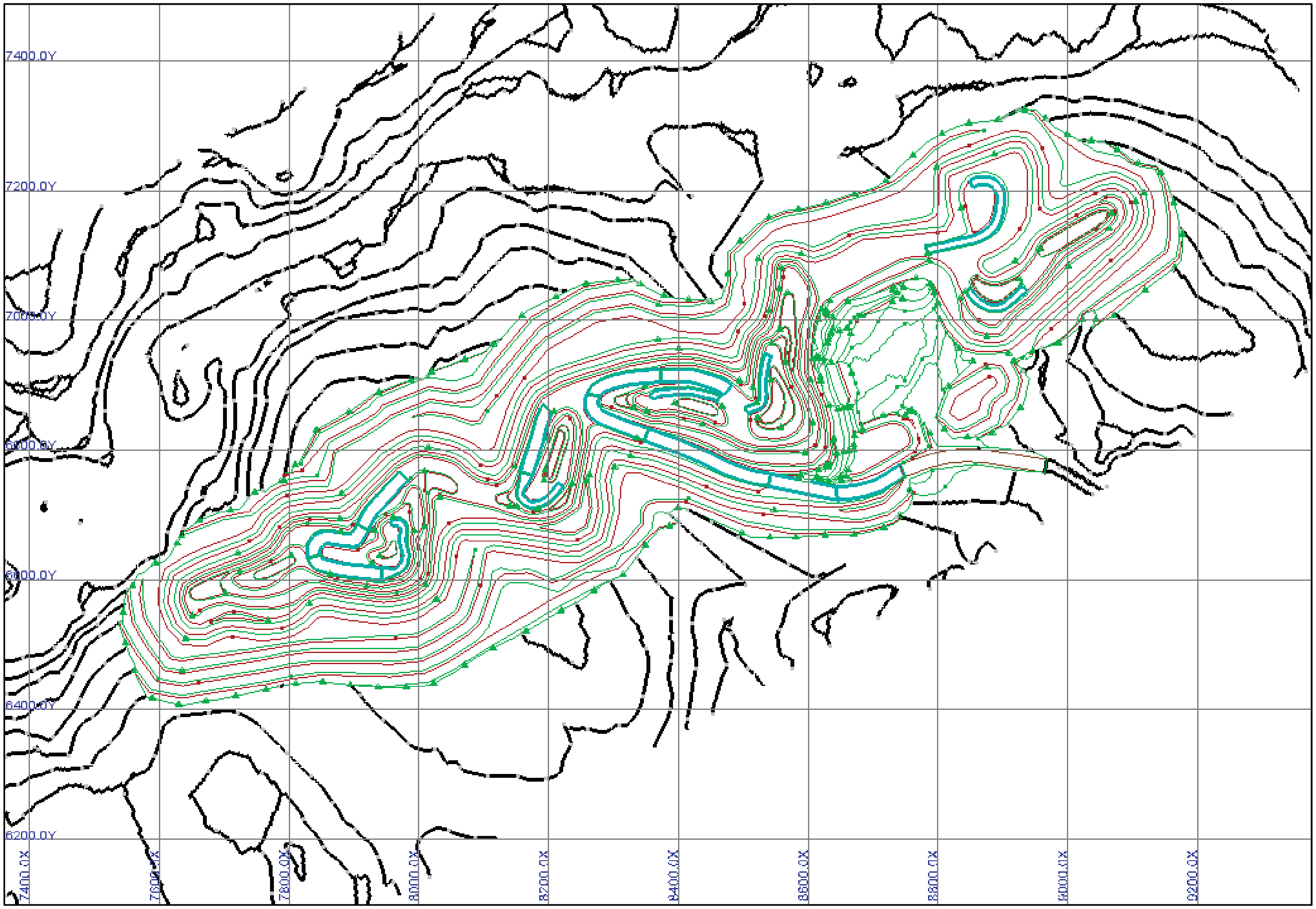
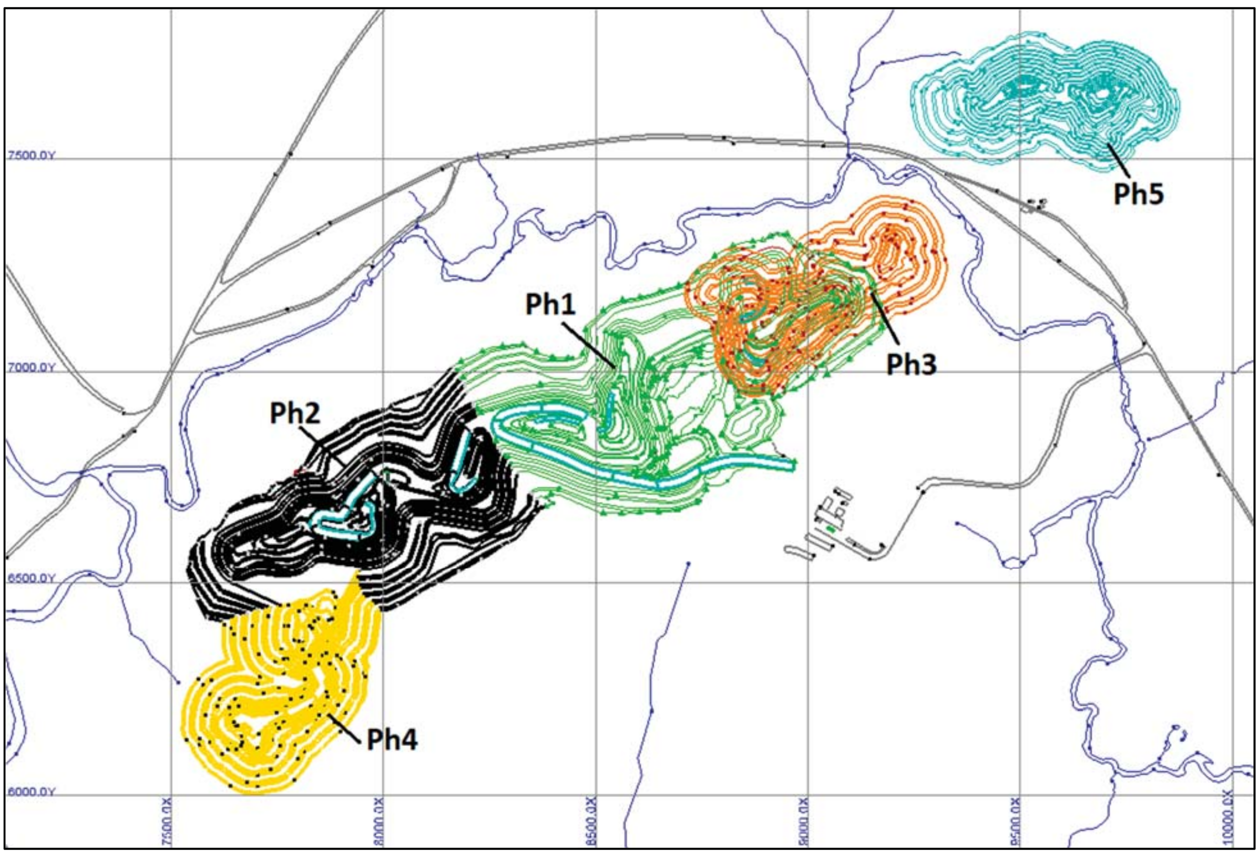


Table 16-2: Pit Tonnage Summary by Phase

Mining Zone	Mill Feed Tonnes	Pb %	Zn %	Overburden Tonnes	Maguma Tonnes	Gypsum Tonnes	Waste Rock Tonnes	Total Waste Tonnes	Total Material Tonnes	Strip Ratio
Phase I	2,225,817	1.10	2.80	13,271,774	494,513	8,743,208	2,180,212	24,689,707	26,915,524	11.1
Phase 2	2,038,429	1.80	3.47	14,360,003	898,905	9,017,636	1,153,820	25,430,364	27,468,793	14.2
Phase 3	349,841	1.20	2.63	1,992,910	0	1,400,000	185,400	3,578,310	3,928,151	10.2
Phase 4	948,762	1.43	1.62	7,800,613	0	31,545	2,042,042	9,874,199	10,823,044	10.4
Phase 5	811,990	2.09	4.12	7,200,826	0	8,208,532	1,244,876	16,654,234	17,466,295	20.5
Total	6,374,839	1.50	3.00	44,626,126	1,393,418	27,400,921	6,806,350	80,226,815	86,601,654	12.6

This preliminary economic assessment is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Figure 16-2: Pit Phases



16.1.4 Waste Dump Criteria

Waste stripped from the open pits will consist of four materials; overburden, gypsum, maguma basement rock, and carbonate waste rock. The plan is to segregate overburden, gypsum, and carbonate waste rock placement within the waste storage area and use the maguma basement rock for road building material as much as possible.

Stantec will use the 2011 waste pile designs for this study as they meet the volume requirements for the projected waste production. The design criteria for the waste piles, including footprint, side slopes, height, setbacks, and drainage were established based on 2011 waste volume requirements, reclamation plan, the Industrial Approval (IA), land ownership constraints, and stakeholder input. Basic configurations include:

- 30 m no disturbance setback distance from the Gays River.
- 50 m setback from any delineated wet land that is not planned to be disturbed.
- 2.25:1 (H:V) side slopes on the waste pile.
- 55 m and 88 m crest elevation levels for the north and south waste piles, respectively.
- Irregular footprint to improve natural aesthetics of final piles.

The two waste piles are shown in Figure 16-3. The mined out areas of the pits will be utilized, when possible, for backfill to reduce out of pit storage of the waste materials while reducing haulage cost.

Figure 16-3: Waste Pile Locations

16.1.5 Open Pit Production Schedule

The production schedule is based on providing an annual mill feed of approximately 852,000 tonnes. Table 16-3 provides an illustration for how the mining operations will sequence through the various pits, pit phases and underground operation. Table 16-4 presents the overall production schedule with annual tonnages of potentially economic mineralization and waste material from all sources.

The underground operation will provide high-grade feed directly to the mill at a rate of 500 t/d, with an average of 294 t/d for the underground production period. This will offset low grade feed from stockpiles which will be delayed until the underground operation is exhausted. At that time, the stockpiles will provide top up the open pit production in order to maintain overall mill feed at 2,600 t/d.

Gypsum will be segregated on the waste piles in order to provide future access should a market be found for it.

Table 16-3: Proposed Mining Sequence

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Phase 1							
Phase 2							
Phase 3							
Phase 4							
Phase 5							
Underground							

Table 16-4: Mine Production Schedule[1]

		Year 1												
Open Pit Mine and Underground Production		Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7	Month 8	Month 9	Month 10	Month 11	Month 12	Sub-Total
Total Open Pit Ore Mined	tonnes	0	14,772	50,604	18,186	74,958	32,122	170,012	108,584	1,129	277,014	61,757	87,223	896,360
Zn Grade	%	0.00	2.32	1.99	1.00	2.06	2.22	2.24	3.58	11.61	2.28	2.89	1.94	2.39
	tonnes	0	343	1,005	181	1,544	713	3,804	3,882	131	6,305	1,783	1,693	21,386
Pb Grade	%	0.00	0.06	0.06	0.18	0.92	0.41	0.52	0.52	4.15	0.40	1.10	1.24	0.59
	tonnes	0	9	28	32	689	131	882	562	47	1,120	680	1,078	5,256
Total Underground Ore Mined	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0
Zn Grade	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0
Pb Grade	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Overburden Dig	tonnes	1,117,291	1,099,274	868,776	1,189,145	788,565	745,738	634,346	482,098	578,811	410,008	1,120,297	528,110	9,562,460
Total Waste Rock	tonnes	0	6,427	53,010	33,448	447,775	26,340	144,751	83,633	7	148,841	46,481	52,937	1,043,651
Total Meguma	tonnes	0	0	0	0	0	0	687	2,923	0	21,659	28,783	1,527	55,579
Total Gypsum	tonnes	0	227,301	116,292	38,767	63,460	573,694	170,033	227,071	552,234	158,696	62,151	560,889	2,750,589
Total all Materials	tonnes	**1,117,291**	**1,347,774**	**1,088,682**	**1,279,545**	**1,374,758**	**1,377,893**	**1,119,830**	**904,310**	**1,132,180**	**1,016,219**	**1,319,469**	**1,230,686**	14,308,638

		Year 2												
Open Pit Mine and Underground Production		Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7	Month 8	Month 9	Month 10	Month 11	Month 12	Sub-Total
Total Open Pit Ore Mined	tonnes	74,041	200,525	48,428	332,049	77,026	84,362	91,993	73,519	71,737	71,658	106,470	144,949	1,376,756
Zn Grade	%	4.88	1.82	2.72	1.46	2.92	3.03	2.43	4.25	2.57	4.03	3.78	4.82	2.86
	tonnes	3,610	3,650	1,317	4,853	2,250	2,553	2,231	3,124	1,844	2,889	4,027	6,991	39,339
Pb Grade	%	1.74	0.33	2.21	0.53	1.13	1.22	0.60	3.77	1.14	1.91	2.75	2.52	1.36
	tonnes	1,286	653	1,071	1,754	873	1,031	550	2,768	817	1,371	2,924	3,659	18,759
Total Underground Ore Mined	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0
Zn Grade	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0
Pb Grade	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Overburden Dig	tonnes	883,674	641,287	531,736	79,238	881,694	343,556	86,849	178,250	176,017	5,761	120,144	63,102	3,991,308
Total Waste Rock	tonnes	23,479	231,976	18,714	368,984	116	264,358	117,980	25,322	32,871	20,204	33,919	44,967	1,182,891
Total Meguma	tonnes	26,703	277	73,768	99,150	0	42,333	11,876	6,032	42,692	42,322	17,186	87,106	449,443
Total Gypsum	tonnes	207,596	13,434	422,406	234,496	216,849	395,165	709,936	944,824	793,209	985,985	776,368	478,144	6,178,412
Total all Materials	tonnes	**1,215,493**	**1,087,499**	**1,095,052**	**1,113,917**	**1,175,684**	**1,129,774**	**1,018,635**	**1,227,946**	**1,116,527**	**1,125,930**	**1,054,087**	**818,268**	13,178,810

		Year 3					Year 4					Year 5	Year 6	Year 7	Year 8	Year 9	LOM
Open Pit Mine and Underground Production		Q1	Q2	Q3	Q4	Sub-Total	Q1	Q2	Q3	Q4	Sub-Total	Sub-Total	Sub-Total	Sub-Total	Sub-Total	Sub-Total	
Total Open Pit Ore Mined	tonnes	192,704	0	303,892	447,084	943,680	292,258	392,409	235,090	218,309	1,138,066	357,275	968,923	829,245	0	0	6,510,305
Zn Grade	%	3.76	0.00	1.75	1.82	2.19	3.09	4.06	5.37	4.74	4.21	2.45	1.57	3.90	0.00	0.00	2.85
	tonnes	7,246	0	5,311	8,124	20,681	9,028	15,949	12,616	10,351	47,945	9,152	15,697	34,165	0	0	188,364
Pb Grade	%	3.55	0.00	0.65	0.52	1.18	1.28	1.71	3.25	2.94	2.16	1.06	1.40	2.01	0.00	0.00	1.56
	tonnes	6,850	0	1,963	2,308	1,112,084	3,753	6,714	7,634	6,424	2,452,572	3,808	13,856	17,331	0	0	3,683,326
Total Underground Ore Mined	tonnes	0	0	0	0	0	0	0	0	0	0	67,403	183,825	38,074	0	0	289,302
Zn Grade	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	6.35	6.52	6.99	0.00	0.00	6.54
	tonnes	0	0	0	0	0	0	0	0	0	0	4,552	12,749	2,835	0	0	20,137
Pb Grade	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	3.66	3.76	4.04	0.00	0.00	3.78
	tonnes	0	0	0	0	0	0	0	0	0	0	2,603	7,296	1,625	0	0	11,525
Total Overburden Dig	tonnes	2,833,055	5,412,357	4,789,628	3,096,529	16,131,569	448,097	117,264	9,702	1,021	576,084	7,959,166	4,085,000	3,268,844	0	0	45,574,431
Total Waste Rock	tonnes	45,096	0	118,924	453,638	617,657	319,503	260,144	73,057	97,317	750,021	2,085,435	1,271,330	0	0	0	6,950,985
Total Meguma	tonnes	135,115	0	0	319	135,434	1,941	94,287	220,702	465,643	782,573	0	0	0	0	0	1,423,028
Total Gypsum	tonnes	1,082,151	201,597	404,576	1,869,643	3,557,967	2,683,124	2,143,886	1,557,928	696,106	7,081,043	32,215	8,382,963	0	0	0	27,983,190
Total all Materials	tonnes	**4,288,121**	**5,613,954**	**5,617,019**	**5,867,213**	21,386,307	**3,744,923**	**3,007,989**	**2,096,479**	**1,478,397**	10,327,788	**10,501,494**	**14,892,041**	**4,136,163**	**0**	**0**	**88,731,241**

1 This preliminary economic assessment is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

16.1.6 Open Pit Equipment Fleet

The open pit mining operations will be performed by contractor mining. All mining equipment will be supplied and operated by the contractor. The quantity and type of equipment needed will be determined by the contractor to meet the production schedule. Support equipment will include a grader, loader, boom truck, pickup trucks, service vehicles, fuel truck, light plants, dewatering pumps, etc. This support equipment will be operated by the owner's employees, on a general basis. Drilling and blasting will be by a drill and blast contractor.

16.1.7 Open Pit Dewatering

The historic operations have, at times, been disrupted by water inflow into the workings. During the 2008 operating period, through a combination of wells and in pit pumping at an average rate of approximately 5,900 gpm, the amount of water was maintained at acceptable levels. The plan is to have sufficient pumping capacity to maintain acceptable water levels in the pit through a combination of in pit and intercept water well operation. Additional back up pumps and where required diesel equipment is also planned.

16.2 Underground Mine

Section 16.2 and associated data is based on previous work by Gerry Beauchamp P. Eng. and independently reviewed with minor commentary updates by Stantec Consulting.

The underground mining operation will target the higher-grade resources located between the Main and Northeast pits, beneath the highway, and beneath the Gays River. The underground workings and related facilities are designed to produce up to 500 t/d of higher grade feed to the mill to blend with the lower grade mill feed from the ongoing open pit operations.

The underground project is based on the Mineral Resources in lenses or “Blocks” 2 and 5. These are the highest grade underground mineral resource areas (see Figure 16-12). Other reasonable underground mining targets have been identified and will be considered for underground mine during mine operations.

The underground mining targets consist of two flat dipping lenses containing zinc and lead mineralization. The target mineralization widths range between 5 m and 12 m, averaging 7 m. The lenses extend to approximately 130 m below surface and the mine area laterally covers a total strike length of about 300 m.

Principal design considerations for this study are:

- Underground mineral resource target containing 331,673 t grading 7.96% Zn and 4.62% Pb. The plan is to produce approximately 283,000 t grading 6.96% Zn and 3.98% Pb from the underground operations;
- Underground and surface infrastructure will be used in conjunction with the open pit mining, where possible, to reduce engineering and building requirements and to improve continuity / flexibility between surface and underground mining; and
- There is a water bearing trench zone occasionally located close or adjacent to the hanging wall contact of the higher-grade mineralization. The trench zone affected previous underground mining operations in the 1980s and 1990s. Dewatering associated with open pit operations adjacent to the underground target is expected to beneficially impact the water table and ease water inflows for the proposed underground mining area. Additional water management techniques will be investigated to mitigate inflows into the underground operation. These may include avoidance of breaking into the trench zone and exposing the underground openings, detailed 3D mapping of the trench zone, advance test drilling to locate the trench zone ahead of mine openings, horizontal drains from the open pits, dewatering wells above the underground operation, and grouting the trench zone in proximity to the underground openings.

16.2.1 Underground Mining Method Selection

A mining method was selected to minimize operating costs and dilution; both of which are often relatively high in this type of deposit. A trade-off study comparing three mining methods was completed to determine the most suitable mining method. These were:

- Conventional Shrinkage
- Modified Cut-and-Fill
- Longhole Stoping

With the Conventional Shrinkage method the mineralization is drilled and blasted by miners working within the stope using handheld drills. During the mining phase blasted mineralization is left in the stope to keep it full and act as a working platform for the miners, and only the swell or surplus graded mineralization is removed from the stope. Draw points are provided at the base of the stope and broken mineralization is drawn from the stope by scooptrams, as required. Once mining of the stope is complete, all the remaining mineralization is removed and the stope is left empty. This method is suited to mining steeply dipping narrow orebodies and is quite selective, but is limited in production capacity, is labour intensive, and is more expensive than other stoping methods.

Modified Cut-and-Fill mining has miners working within the stope using electric hydraulic two boom jumbo drills. This method requires backfill, which for the ScoZinc project would consist of waste development and a surface stockpile from open pit mining. The mineralization would be loaded by scooptrams into trucks for haulage to surface. This method is quite selective, but like Shrinkage Stoping, is limited in production capacity, is labour intensive, and is higher cost than the Longhole method. The advantage of Cut-and-Fill over Shrinkage is that the backfill will provide almost immediate support to the hanging wall and footwall as the stope is being mined.

The advantage of both of these methods is their selectivity and lower dilution in comparison to Longhole mining.

While Longhole Stoping offers the best potential for high production rate, high workforce productivity, and lower operating costs there are a number of concerns with this method. First and foremost is the potential for increased dilution. Dilution is the failure of stope walls due to geotechnical conditions. A key geotechnical feature of the deposit is the presence of weak hanging wall and footwall rock, creating a risk of wall slough breaking out from the stope walls. Control of this risk would require appropriate stope design and ground support. While this risk applies to all three mining methods, it is most relevant with Longhole mining. Cut-and-Fill will result in the least wall slough as the stope walls are supported throughout the mining cycle by the backfill. With shrinkage, mining the stope walls are supported by the broken ore during the mining cycle, but wall sloughing can occur during the pull-down phase. A big disadvantage with Shrinkage is that for this method to be practical, the stopes need to be sized a minimum of 75 m long by 50 m high. Based on the rock mechanics analysis, stopes of this size would experience major wall sloughing. With open Longhole Stoping, wall slough is the major concern.

Considering the above discussion, Cut-and-Fill mining is selected as the preferred option. For all mineralization included in mining Blocks 2 and 5, variations of Cut-and-Fill mining will be utilized as the primary mining method. Cut-and-Fill will be employed in areas with mineralization horizontal thickness of less than 7 m, while modified Cut-and-Fill will be utilized for mineralization thickness over 7 m. A variation will be used to recover the sill pillars created for the multiple workings.

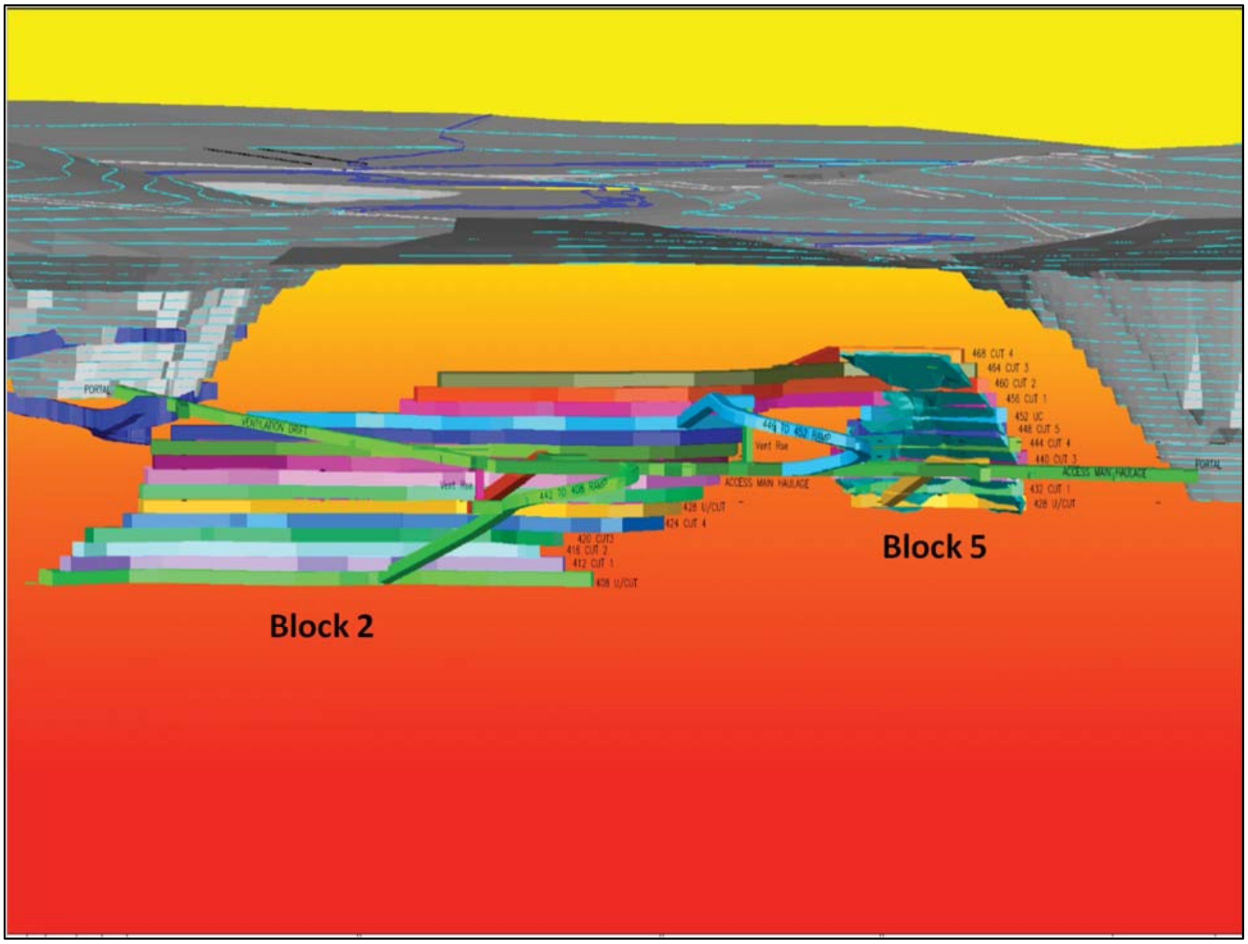
16.2.1.1 Vertical Sequence

The current mining plan calls for bottom-up mining from one horizon of Block 2 and one horizon of Block 5 (Figure 16-4).

16.2.1.2 Horizontal Sequence

At each lift, crosscuts will typically be driven to the middle of the mineralization and mining then will take place in a centre-out fashion. The sill drift will be driven along the footwall to establish the mineralization/waste contacts. In areas where the mineralization is greater than 7 m in width, 5 m wide cross cuts will be driven from the footwall contact drift to the hanging wall contact. Each drift will be separated by a 5 m wide pillar, hence the modified Cut-and-Fill (Figure 16-4).

Figure 16-4: General 3D View of Underground Mining Blocks and Planned Development



16.2.2 Access System

16.2.2.1 Mining Areas

The underground mining target is divided into two mining blocks as shown in Figure 16-4:

- Block 2 Mining Zone - Extends from the bottom of the Main open pit to a depth of approximately 130 m and a strike length of approximately 300 m. Only the higher grade upper part of Block 2 is included in this mine plan.

- Block 5 Mining Zone – Adjacent to the Northeast pit, extending from about mid pit elevation to about the bottom of the Northeast open pit and has a strike length of approximately 80 m.

16.2.2.2 Mine Access System

The underground mining targets will be accessed by a 4.5 m wide by 4.5 m high ramp grading 15% and designed to accommodate 32-tonne haulage trucks. The portal as well as all access development is located in the footwall of the orebody. From the surface portal, a single ramp will extend to the 439 m level from which point it will split into two ramps one to provide access to Block 2 and the second ramp leading to Block 5. Each of the two mining zone access ramps are centrally located to the resource zone, and spiral to the bottom and top of the zone, providing access to the various entrances to the Cut-and-Fill workings as shown in Figure 16-13.

The stoping areas for each mining zone will be accessed by 4.5 m wide x 3.5 m high crosscuts to the mineralization from the ramp for that zone. The access point will be midway along the strike length of that zone which will provide two headings producing equal quantities of material over a similar period of time. The vertical spacing of the access cross cuts will be at 25 m intervals. Typical drawings of the main ramp and access cross cuts are shown in Figure 16-5, Figure 16-5, and Figure 16-7.

Figure 16-5: Plan View of Accesses to Underground Mining Blocks

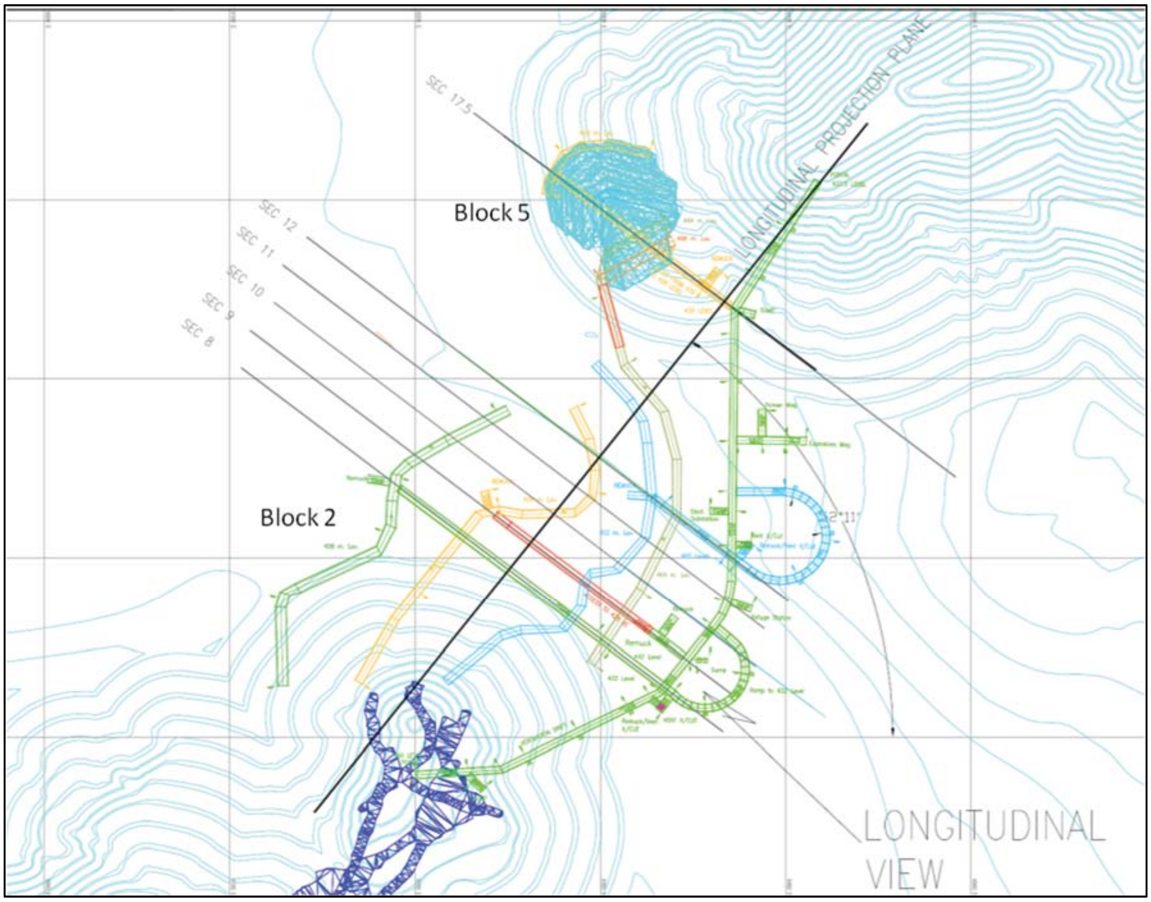


Figure 16-6: Plan View of Ramp from Surface to 468 Level

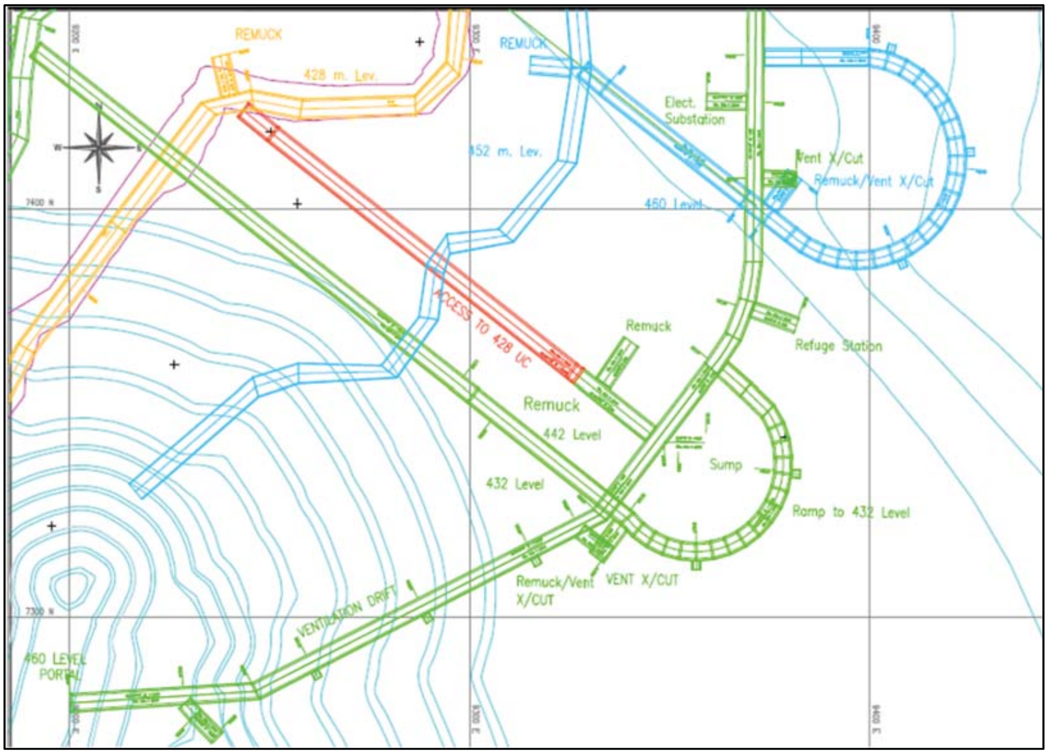
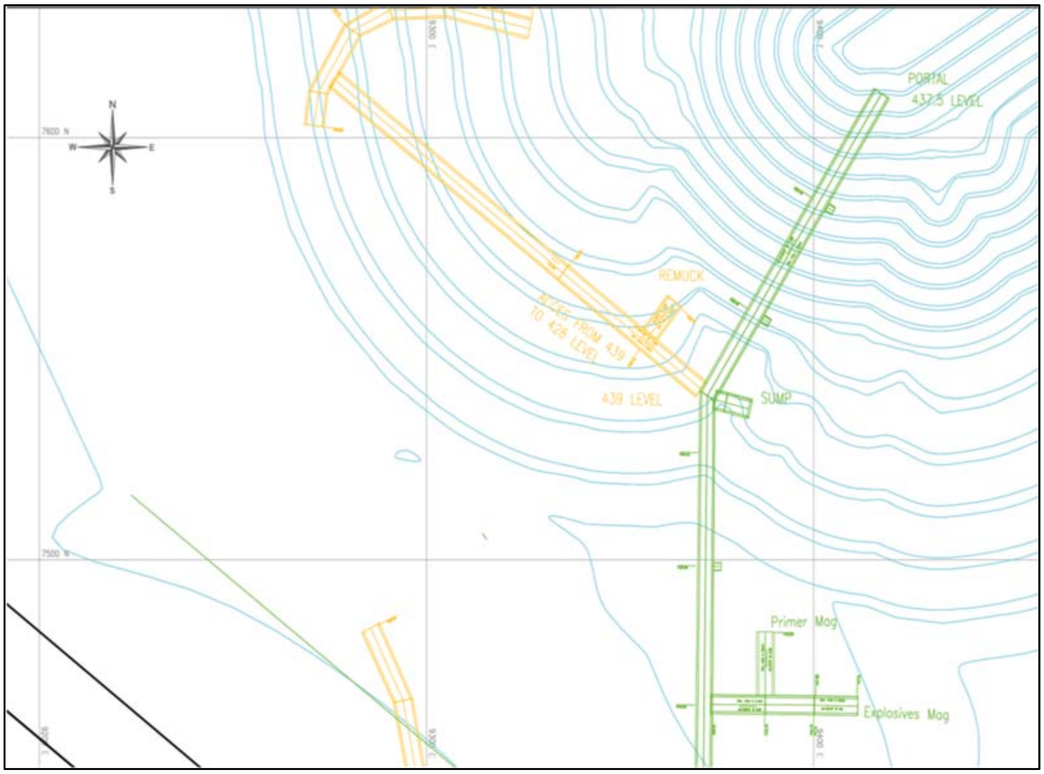


Figure 16-7: Plan View of Ramp from Surface to 439 Level

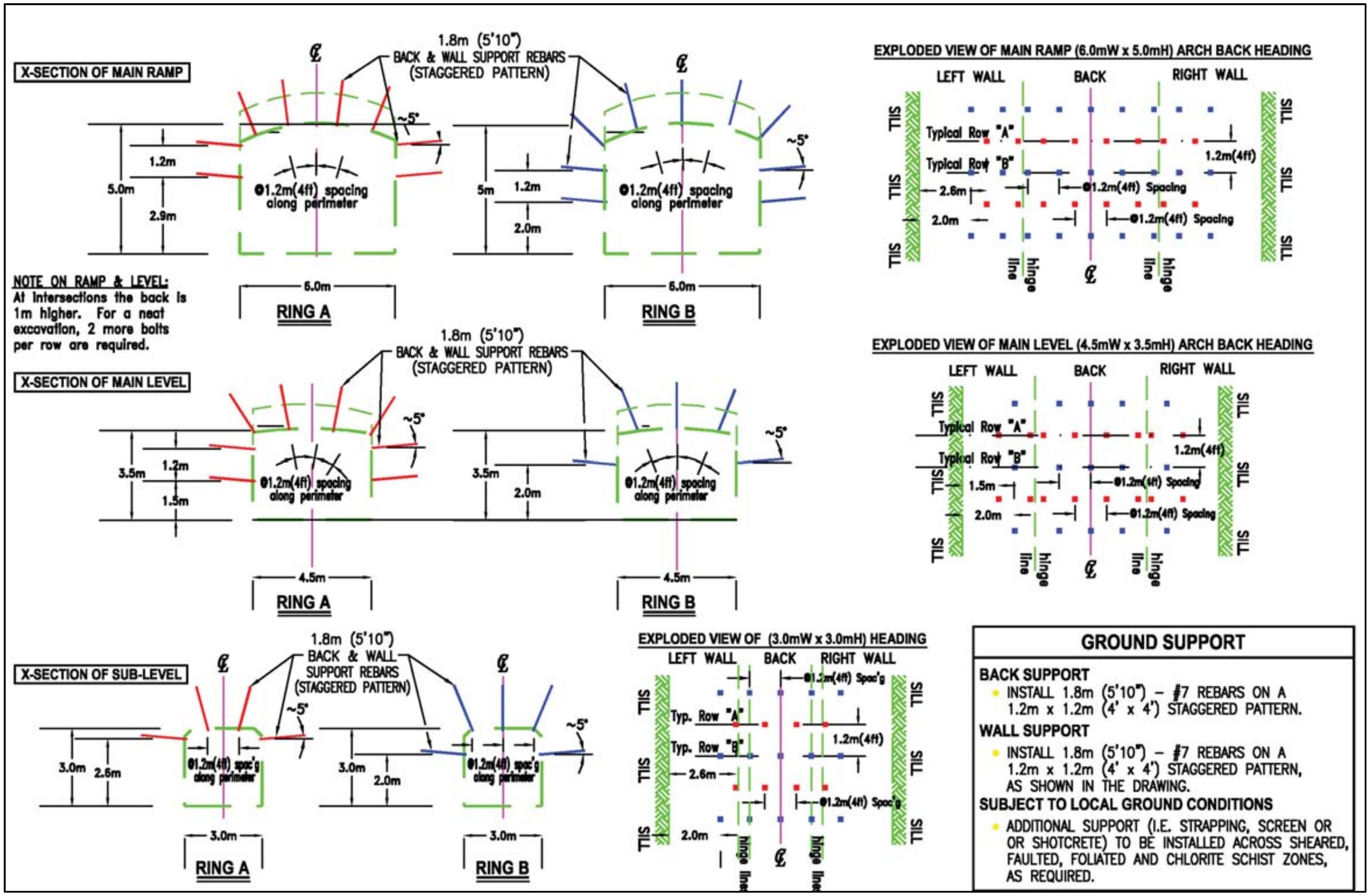


16.2.3 Services and Underground Support

The main haul ramps will be equipped with a 150 mm airline; a 150 mm discharge line and a 75 mm water line. Stope access crosscuts will be equipped with a 50 mm airline, a 50 mm discharge line and a 50 mm water line. Cut-and-Fill lifts in ore will be equipped with a 50 mm airline and a 50 mm water line. Ventilation ducting will be installed in all headings until connection can be made to the central ventilation exhaust system.

Primary support: 2.2 m (7 ft 4 in) long – No. 6 or No. 7 resin rebars (with 5 in x 5 in x ¼ in plates) on a 1.2 m x 1.2 metre (4 ft x 4 ft) pattern. The pattern to be extended across the back and down the walls to within 1.8 m (6 ft) of the sill. Generally speaking, this can be applied to a drift span of 7.0 m if no major adverse structures are encountered. Rebar bolts will be installed using electric-hydraulic roofbolters or stopers and jacklegs. Cut-and-Fill lifts will be bolted using the same 2.2 m resin rebar set bolts in the back and walls. Screen and cable bolts may be used as required. Typical ground support requirements are shown in Figure 16-8.

Figure 16-8: Standard Ground Support



16.2.4 Details of Stoping System

16.2.4.1 Stope Design

Mechanized Cut-and-Fill mining is the method selected for the project and it has been carefully adapted and designed to suit the requirements of the ScoZinc underground project, in particular those of a shallow dipping vein, and geotechnical constraints which limit the length of unsupported back and hanging wall spans. This method was also selected in order to use uncemented backfill, thus minimizing operating costs and benefiting the project economics.

The typical Cut-and-Fill lift size will be 5 m high by ore width (average of 7.0 m) and 300 m along strike. The drift height may have to be adjusted down depending on the thickness of the ore and, to keep dilution to a minimum, it may be required to first drill, blast and muck the mineralization then slash the drift to the size required.

Removal will be done directly from mucking blasted breasts, and sill mucking of the completed lifts. The mining of the Cut-and-Fill lifts must be advanced along strike, from the central access crosscut from the main ramp to the extremity of the mining zone. Because of the water bearing trench zone in the hanging wall, three test holes three drill steels in length will be drilled with all rounds of Cut-and-Fill breasts in mineralization. This will reduce the danger of breaking into the trench zone when blasting

The modified Cut-and-Fill mining involves drifting along strike in the ore zone, following the footwall “ore”/waste contact using two boom jumbos. Cross cuts will be driven from the footwall drift to the opposite wall (i.e. from footwall to hanging wall,). The drifts will be separated by 5 m wide rib pillars (zone dependant). When drifting is complete, services (pipe, vent duct, power cables) will be stripped from the heading. The heading will be backfilled to within 0.5 m to 1 m from the back, using five yard scooptrams with ejector buckets. Following backfilling, the entrance will be back-slashed at +15% to establish a face for the next cut. The mining cycle is then repeated.

A typical stope section and plan are shown in and Figure 16-10. Cross sections of stopes and accesses are shown in Figure 16-11 through Figure 16-14.

Figure 16-9: Typical Stope Section

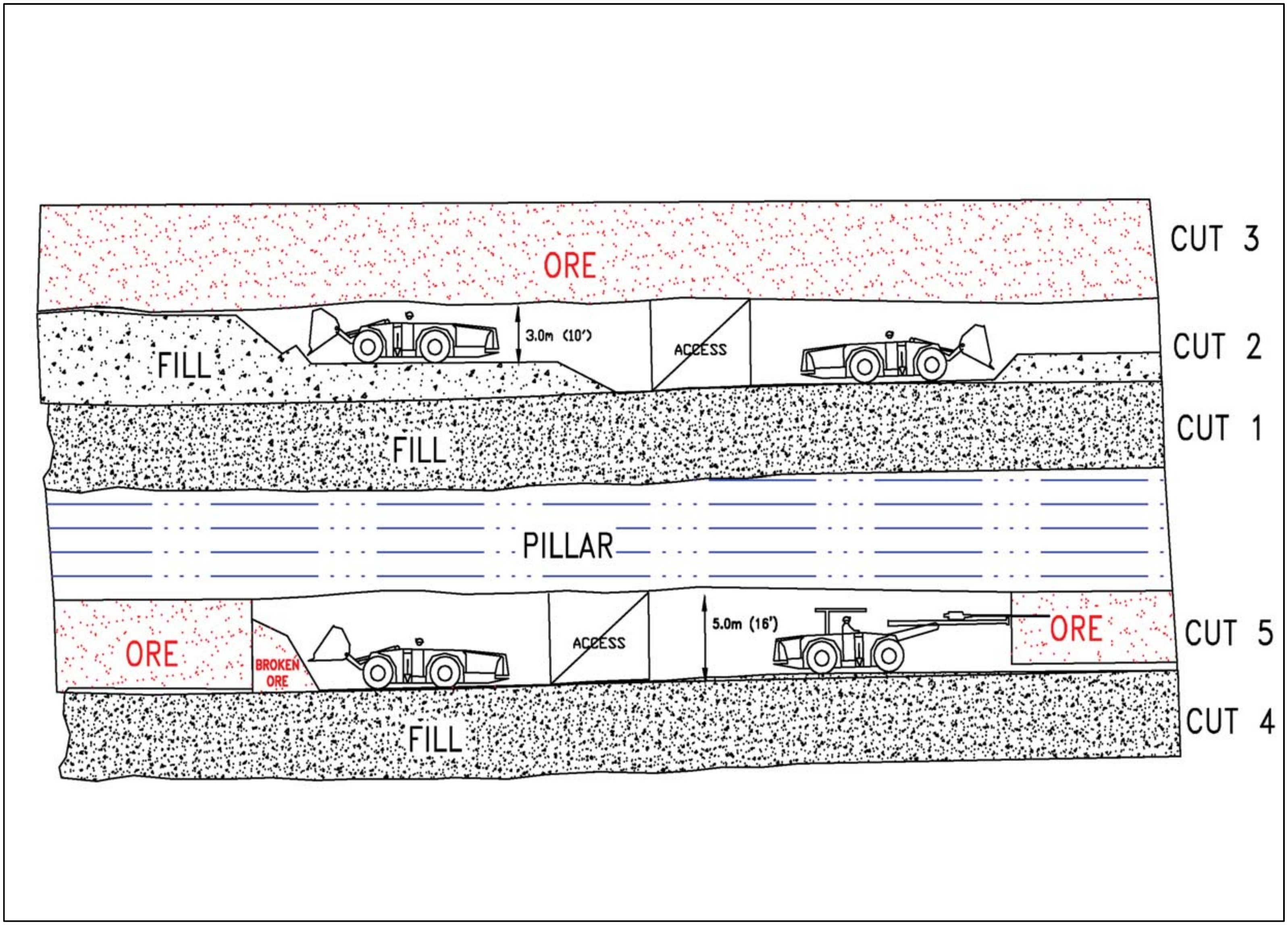


Figure 16-10: Typical Stope Plan

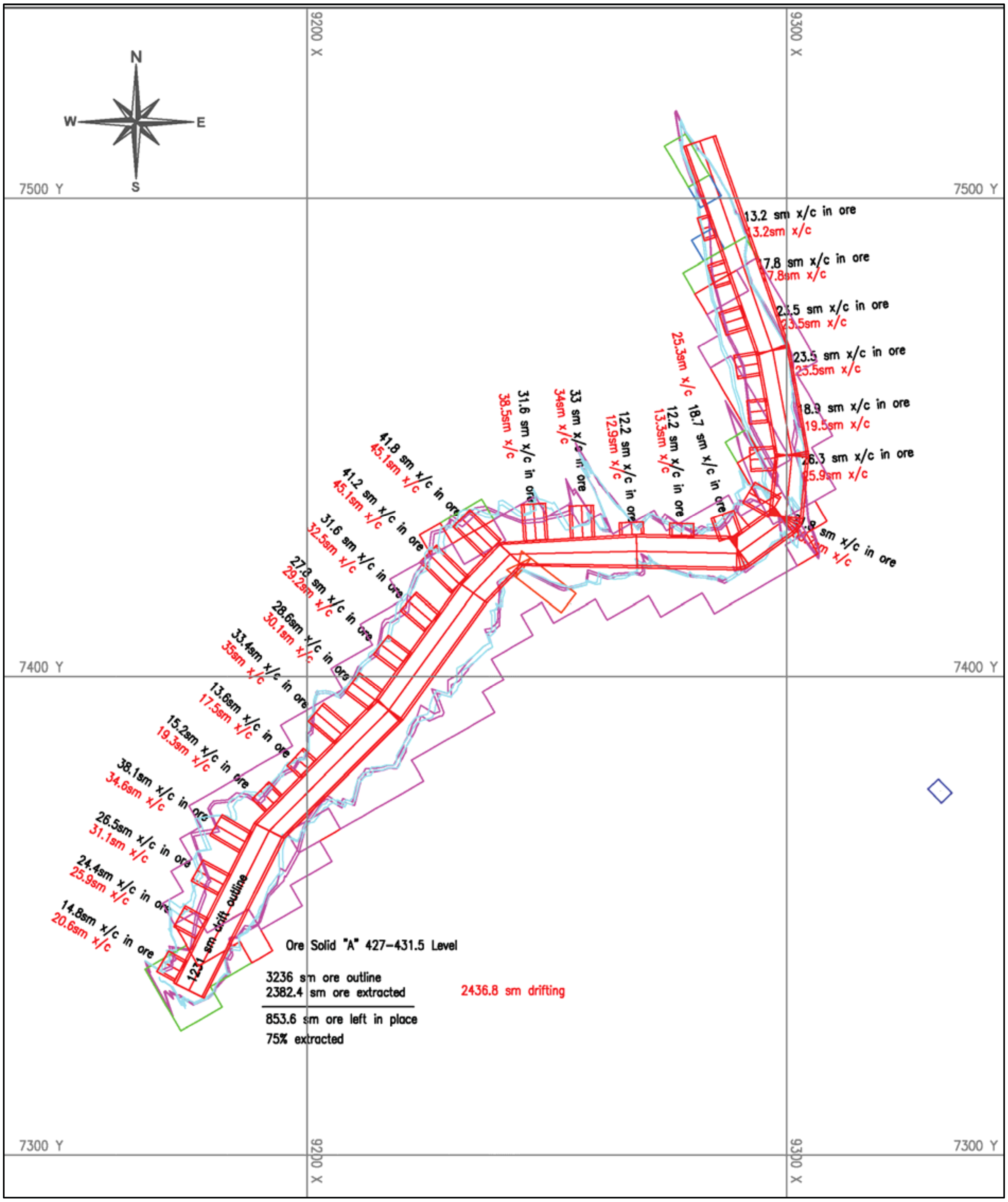


Figure 16-11: Cross Section 8

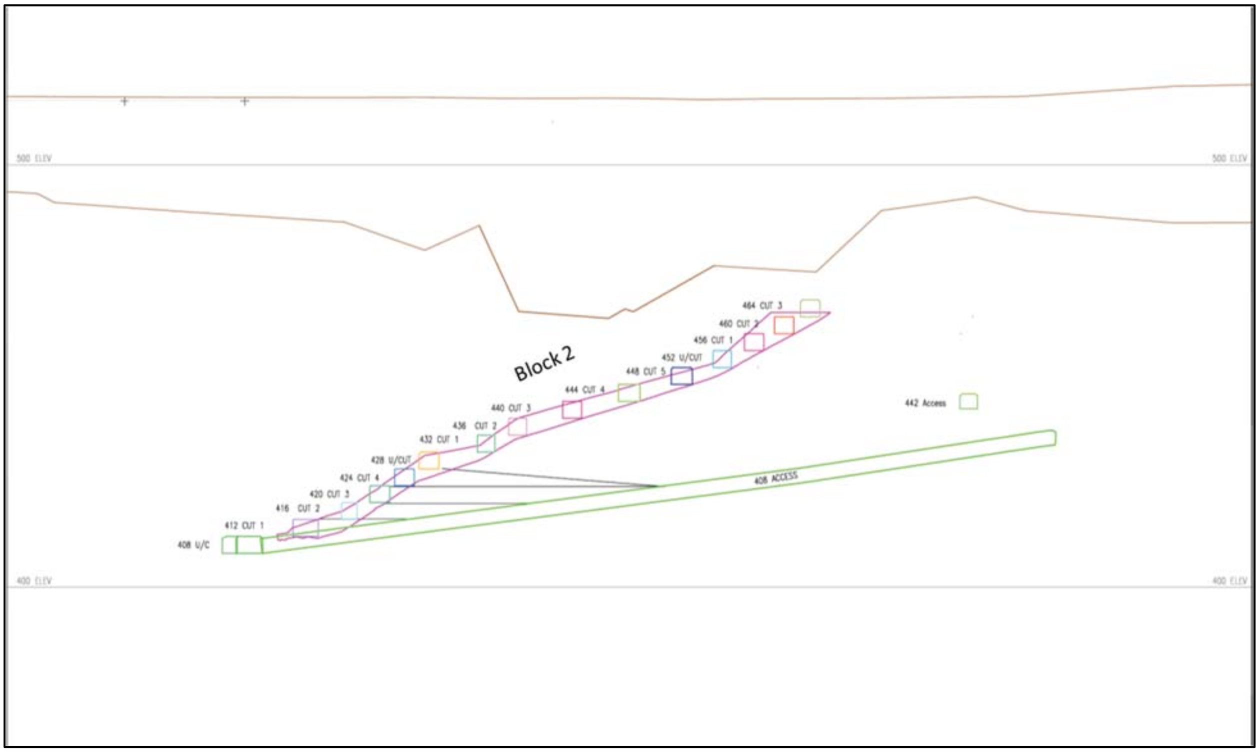


Figure 16-12: Cross Section 9

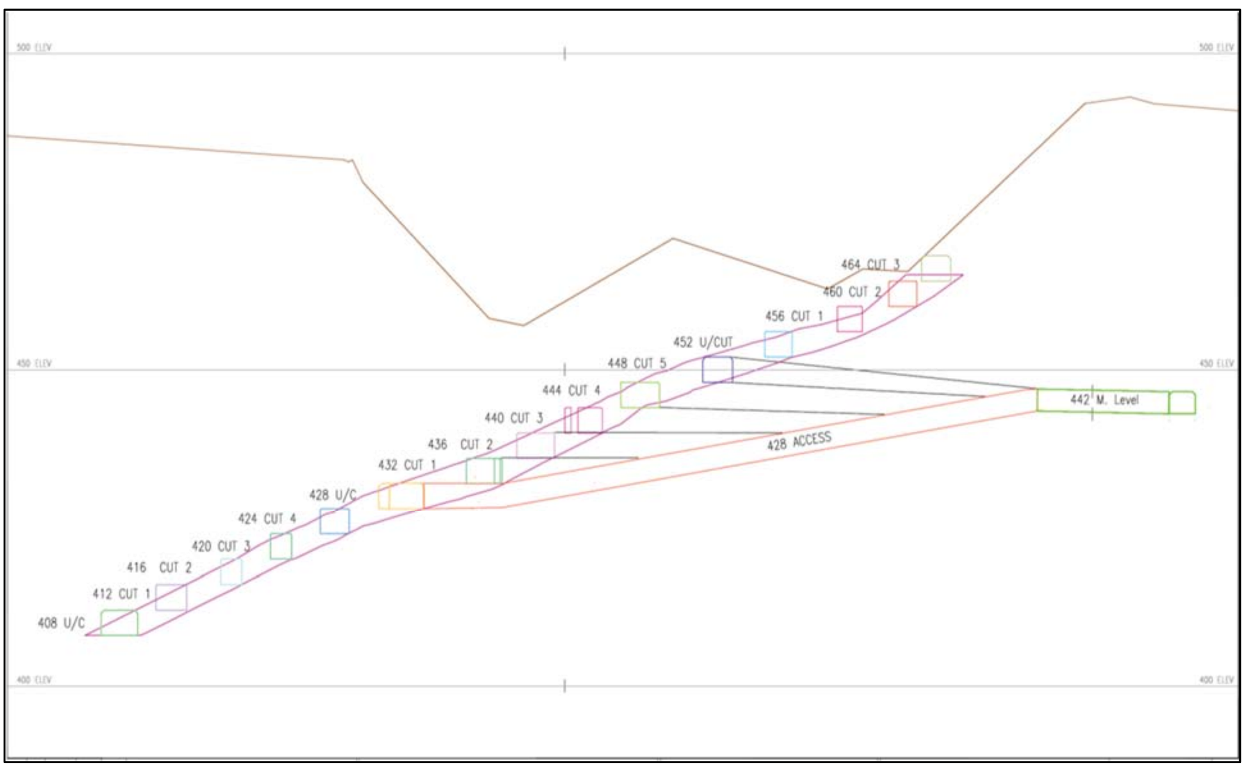


Figure 16-13: Cross Section 12

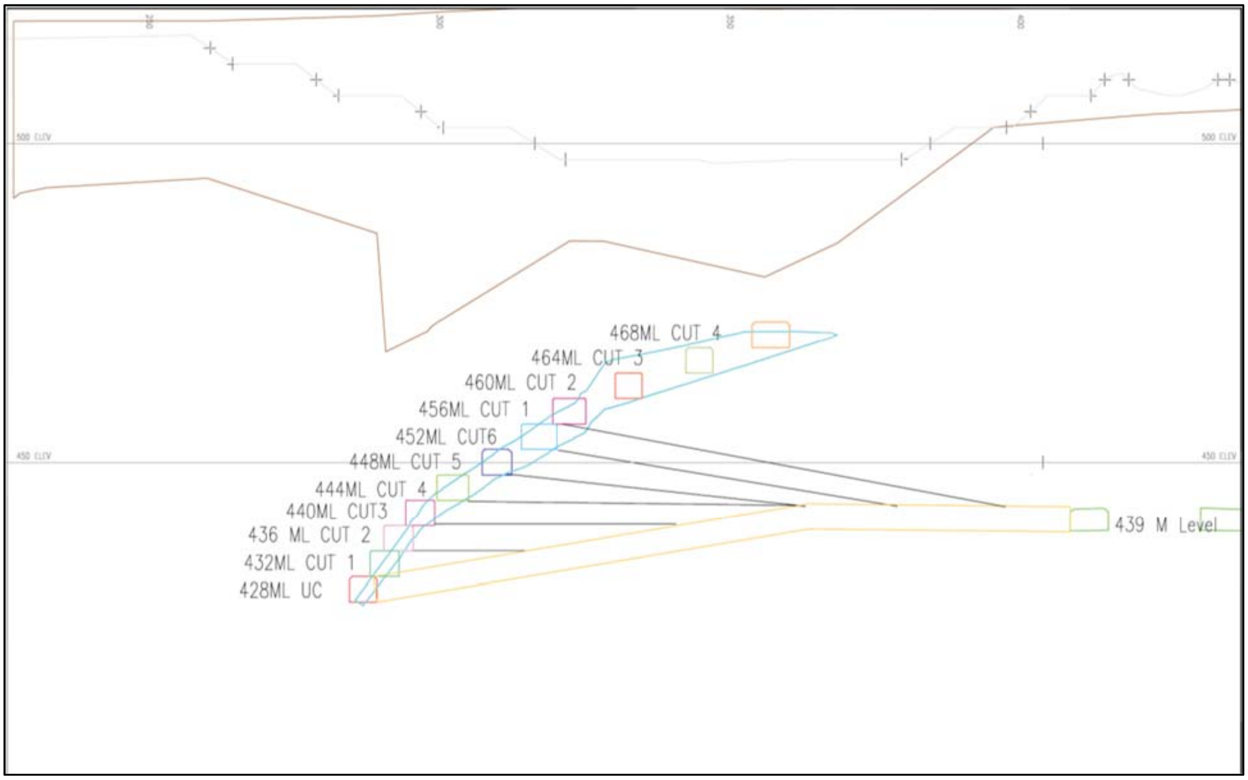
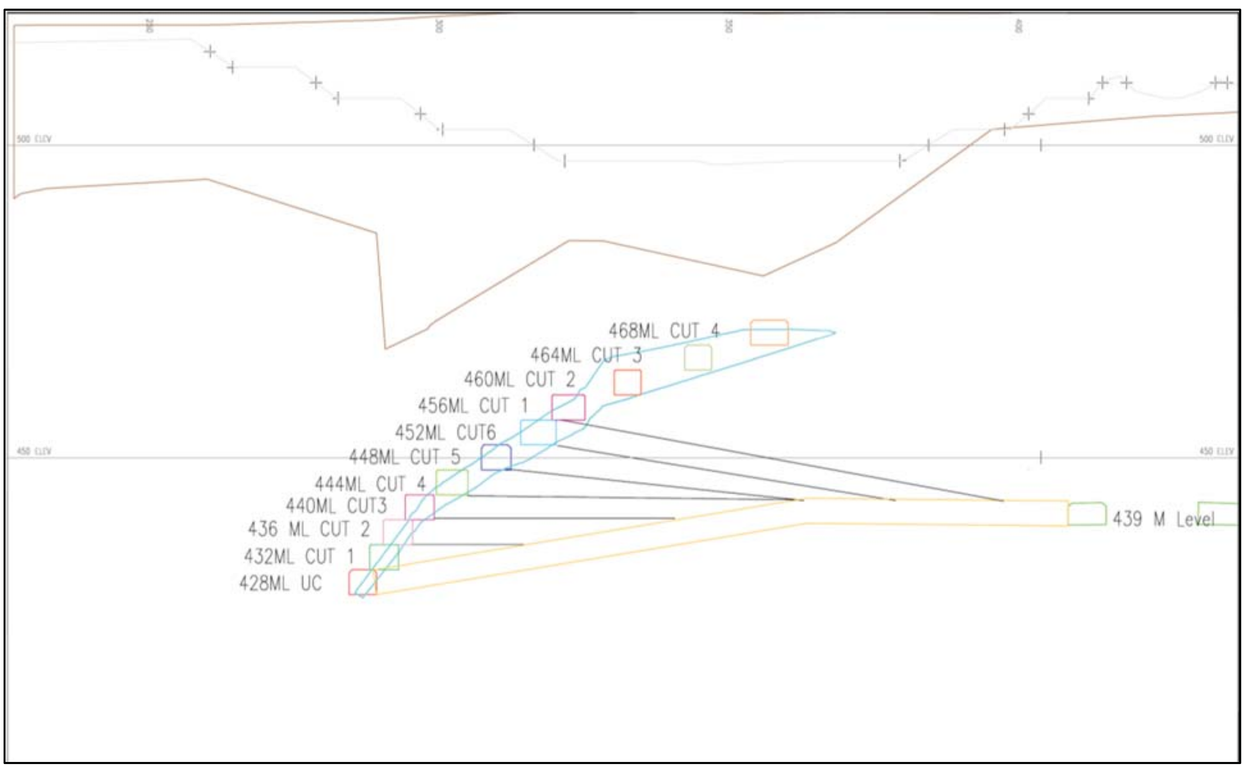


Figure 16-14: Cross Section 17.5



16.2.4.2 Stoping Operations

Strict geological and engineering control will be required to minimize dilution and maximize recovery. This will include detailed geological mapping and sampling of the Cut-and-Fill lifts which, in combination with diamond drill hole data, will be used for the detailed ore projections between the stoping blocks. If low grade areas are identified, the design pillar locations will be reviewed to determine if they can be moved to into the low grade and improve mining recovery.

16.2.4.3 Raise Development

Two short conventional raises will be driven from the 442 m level to the 460 m level (about 15 m) and from the 432 m level to the 442 m level (about 8 m). The ventilation raises will be equipped with manways for a secondary egress from the mine.

16.2.4.4 Crown Pillar

It is necessary to leave a crown pillar to provide a safe barrier between the top of mining Blocks 2 and 5 and the bottom of the planned open pits. There is evidence from the diamond drilling that the bedrock is weathered close to surface. With a maximum mining opening expected of 5.5 m and considering some possible sloughing to 6.0 m wide, then with a rule of thumb height to width ratio of three, a crown pillar of 18 m vertical would be suitable. This will need to be verified through a detailed geomechanical investigation prior to final design.

16.2.4.5 Backfill

The purpose of backfill is not to transmit the rock stresses, but to provide confinement to the rock mass so the rock itself will retain a load carrying capacity and will improve load shedding to the abutments. This leads to less deterioration in ground conditions in the mine, improving the safety and the economics of the mining operations. With the Modified Cut-and-Fill method the pillars remain inside the stope to support the back. The mined out stopes can be backfilled with un-cemented fill; tight filling is not necessarily required but will improve recovery of broken material on subsequent cuts. Pillars are extended through several layers of fill and the fill is contributing to the pillar supporting ability. Backfilling of the Cut-and-Fill stopes is planned on an ongoing basis for use as support. Waste produced from development will be placed in mined out stopes, versus hauling waste to surface, whenever possible. 5 yard scooptrams using ejector buckets will be used to place the waste fill in the stopes.

Mine waste rock brought to the surface will be segregated according to its acid generating potential and stored in dedicated stockpiles, although there is no history of acid generating material at Gays River. Potentially acid generating mine rock, if encountered, will be returned underground for use as backfill, none will remain on the surface upon completion of mining activities. Non-acid generating rock will be used for construction or will be returned underground as backfill. Any additional backfill required will be from a surface stockpile created by the open pit mining.

16.2.5 Mineable Resource

As this report is a Preliminary Economic Assessment there are no declared mineral reserves. The Mineable Resources for the underground operation were estimated based on the higher-grade Mineral Resources targeted for inclusion in the mining plan with appropriate application of mining dilution, recovery and economic factors. The Mineable Resources are summarized in Table 16-5

Table 16-5: Underground Mineable Resources

Mining Zone	Tonnes	Zn Grade (%)	Pb Grade (%)
Block 2	284,078	8.53	4.95
Block 5	47,595	4.56	2.63
Total	331,673	7.96	4.62

16.2.5.1 Dilution

Mining dilution is waste rock that is mined with the ore and cannot be separated out prior to transport to the concentrator. The dilution can be planned, which is waste included in the design to make it practical and efficient, or unplanned which is waste mined due to overbreak (mining outside the plan limits) or sloughing from the back and walls due to geotechnical reasons.

For Cut-and-Fill mining, planned dilution was estimated by specifying a minimum mining width of 4.0 m (13 ft). The minimum mining width was chosen based on the mining method, planned equipment, length of exposed hanging wall and horizontal width of the exposed back. Unplanned dilution due to overbreak and hanging wall sloughing was estimated at approximately 10% to all Cut-and-Fill lifts.

A second source of dilution will be mucking dilution. This dilution will be the mucking of some backfill waste while the final mucking of the Cut-and-Fill lift is done. The resulting overall dilution was estimated to be 17.0%.

16.2.5.2 Mining Recovery

Mining recovery is the recovery of the mineral resources included in the mine plan and does not apply to those mineral resources already excluded as being outside the mine plan. Recovery losses result from resources left behind in pillars, ore left in the stope during sill mucking, and ore that does not meet specified economic criteria.

The resources left unmined in pillars was estimated based on the stope layout design as previously described. In total, it has been estimated that 27% of the mineral resources included in the mine plan will be lost as a combination of non-recoverable sill pillars and stope mucking resulting in an overall mining recovery of 73% (see Table 16-6).

Following the application of mining recovery to the diluted resources contained within the mine plan, the underground Recoverable Mineral Resources were estimated and are presented in Table 16-6.

Table 16-6: Underground Recoverable Mineral Resources

Mining Zone	Diluted Resource (tonnes)	Mining Recovery			Recoverable Mineral Resource		
		Mining (%)	Pillars (%)	Overall (%)	Tonnes	Zc Grade (%)	Pb Grade (%)
Block 2	332,371	75	98	73	242,631	7.45	4.27
Block 5	55,686	75	98	73	40,651	4.06	2.28
Total	388,057	75	98	73	283,282	6.96	3.98

16.2.6 Development Schedule

16.2.6.1 Development

Mine and stope development totals are summarized in Table 16-7.

Table 16-7: Capital and Operating Development

Development Type	Width × Height (m)	Ore / Waste	Length (m)
Capital			
Ramp/Remucks/Sumps, etc	4.5 x 3.5	Waste	980
Raises and Manways	2.7 x 2.7	Waste	46
Sub Total			**1,026**
Operating			
Drifts and Cross Cuts	4.5 x 3.5	Waste	250
Backslashing/Crosscuts/Remucks	4.5 x 4.5	Waste	834
Drifts and Cross Cuts	4.5 x 4.5	Ore	293
Raises and Manways	2.7 x 2.7	Waste	126
Sub Total			**1,377**
Total Mine			
Ramp/Remucks/Backslashing	4.5 x4.5	Waste	1,814
Raises and Manways	2.7 x 2.7	Waste	46
Drifts and Cross Cuts	4.5 x 3.5	Waste	250
Drifts and Cross Cuts	4.5 x 4.5	Ore	293
Total			**2,403**

16.2.6.2 Development Schedules

All development will be completed by ScoZinc personnel. Initially the ramp and lateral development will be developed at an advance rate of 4.0 m (1.0 round) per day. Once the ventilation raise has been completed to surface, waste development will be accelerated to 6 m per day until the entire mine has been developed. Conventional raise development is scheduled at 2.2 m per day. A summary of the mine development cost by type of development is presented in Table 16-8 and a table of waste produced is displayed in Table 16-9.

Table 16-8: Development Cost by Category

Development Type	Ore / Waste	Cost ($ x 1,000)
Capital		
Ramp/Remucks/Sumps, etc	Waste	3,108
Conventional Raises	Waste	18
Raises and Manways	Waste	10
Sub Total		**3,136**
Operating		
Drifts and Cross Cuts	Waste/Ore	1,466
Backslashing/Crosscuts/Remucks	Waste	2,250
Definition Drilling Drifts	Waste	54
Sub Total		**3,770**
Total Mine		
Ramp/Remucks/Backslashing	Waste	3,108
Drifts/Backslashing/Crosscuts/Remucks	Waste/Ore	3,716
Conventional Raises	Waste	28
Definition Drilling Drifts	Waste	54
Total		**6,906**

Table 16-9: Development Waste

Development Type	Capital Period (tonnes)	Operating Period (tonnes)	Total (tonnes)
Drifts and Ramps	39,332	49,285	88,618
Raises	851	2,333	3,184
Total	40,183	49,285	91,801

16.2.7 Production Schedule

All production will come from Cut-and-Fill stope mining. Production is scheduled at a rate of 180,000 tonnes per year, (500 t/d, 360 days per year) with 242,631 t scheduled from Block 2 and 40,651 tonnes from Block 5. Production from the underground operation will go directly to the mill and be blended with lower grade feed from the open pits and stockpiles (see Table 16-4).

Cut-and-Fill mining will produce on average one 4.0-m-long by 5.0-m-wide by 5.0-m-high Cut-and-fill round per day, or approximately 280 tonnes per day. Mining areas are assumed to be in the mining cycle 75% of the time, and in the backfill cycle or otherwise unavailable for mining 25% of the time. The annual productivity of a single

Cut-and-Fill area will be 210 tonnes per day. To attain the scheduled production rate of 500 tonnes per day the mine is planned to have three mining areas producing ore.

16.2.8 Underground Mobile Equipment

Haulage ramps, stope access cross cuts, and Cut-and-Fill mining will be done using two-boom electric hydraulic jumbos, 3.8 m^3 (5 yd^3) scooptrams and 32 tonne trucks. Waste development will be hauled to remuck bays where it will be loaded onto 32 t trucks by scooptrams and either hauled to surface or used to backfill mined-out Cut-and-Fill lifts. Mineralized material will be hauled to remuck bays located in the stope access crosscuts, where it will be loaded onto 32 t trucks by scooptrams and hauled to surface. Run-of-mine mineralization will be trucked via the mine ramp to surface and dumped on a stockpile near the portal. A summary of underground equipment required is presented in Table 16-10.

Table 16-10: Underground Mobile Equipment

Category	Number of Units	
	Capital	Operations
Diesel Equipment		
Scooptram 3.8 m^3 (5 yd^3) c/w ejector buckets	3	
Underground Trucks 32 Tonne	2	
RBM2D Two Boom Jumbo	2	
Bolter	1	
ANFO Truck c/w Basket	1	
Scissor Lift Truck	1	
Personnel Carriers Toyota	3	2
Non Diesel Equipment		
Shotcrete Machine - Dry Mix		1
Ventilation Fans 48" - 150 hp		5
1,500 cfm Compressor c/w Tank	1	
U/G Submersible Pumps – 13 hp	3	3
U/G Submersible Pumps – 30 hp	2	
U/G Submersible Pumps – 140 hp	1	1
Support Equipment		
Mine Rescue equipment	1	
Surveyor's Equipment	1	
Grout Pump and Mixer	1	
Portable Electrical Substations 1000 KVA	1	1
Hand Held Drills - Stopers		4
Hand Held Drills - Jacklegs		4
Ventilation bulkheads LOM		5
Refuge Stations		3
Main Vent Fans, Heaters	1	
Surface Buildings	1	
Main Dewatering Pump Station		1

16.2.9 Underground Fixed Equipment

Underground installations will be mobile including such items as electrical substations. Permanent installations will include main mine dewatering system, refuge stations, and ventilation bulkheads. The requirements for these units of equipment are included in Table 16-10.

16.2.10 Workforce

The underground mine is planned to be developed and operated by ScoZinc personnel. Initial development of the project will take 12 months at which time ore from the underground operation will be processed in the mill.

ScoZinc will be responsible for all technical services including geology, mine planning, surveying and supervision of the mining personnel for scope of work and assurance that all mining standards and safety procedures are being diligently adhered to.

ScoZinc will provide all the workforce, equipment and facilities required for the development of the project and LOM resource production. The workforce to be provided includes miners, direct supervision, on-site safety supervisor, first aid attendant, training services and other support staff. Contractors will be used as required for specialty work such as driving Alimak raises and diamond drilling.

16.2.10.1 Mining Workforce

The workforce required for the underground mining over the LOM from start of project to end of production is shown in Table 16-11.

Table 16-11: Workforce

Category	Personnel
Staff	
Mine Foreman	1
Shift Boss	4
Safety/Training	1
Senior Engineer	1
Planning Engineer	1
Surveyor	1
Geologist	1
Timekeeper/Clerk	1
Subtotal Staff/Support	11
Mining	
Lead Mechanic	1
Mechanic	11
Electrician	2
Surface Equip operator/Dryman	1

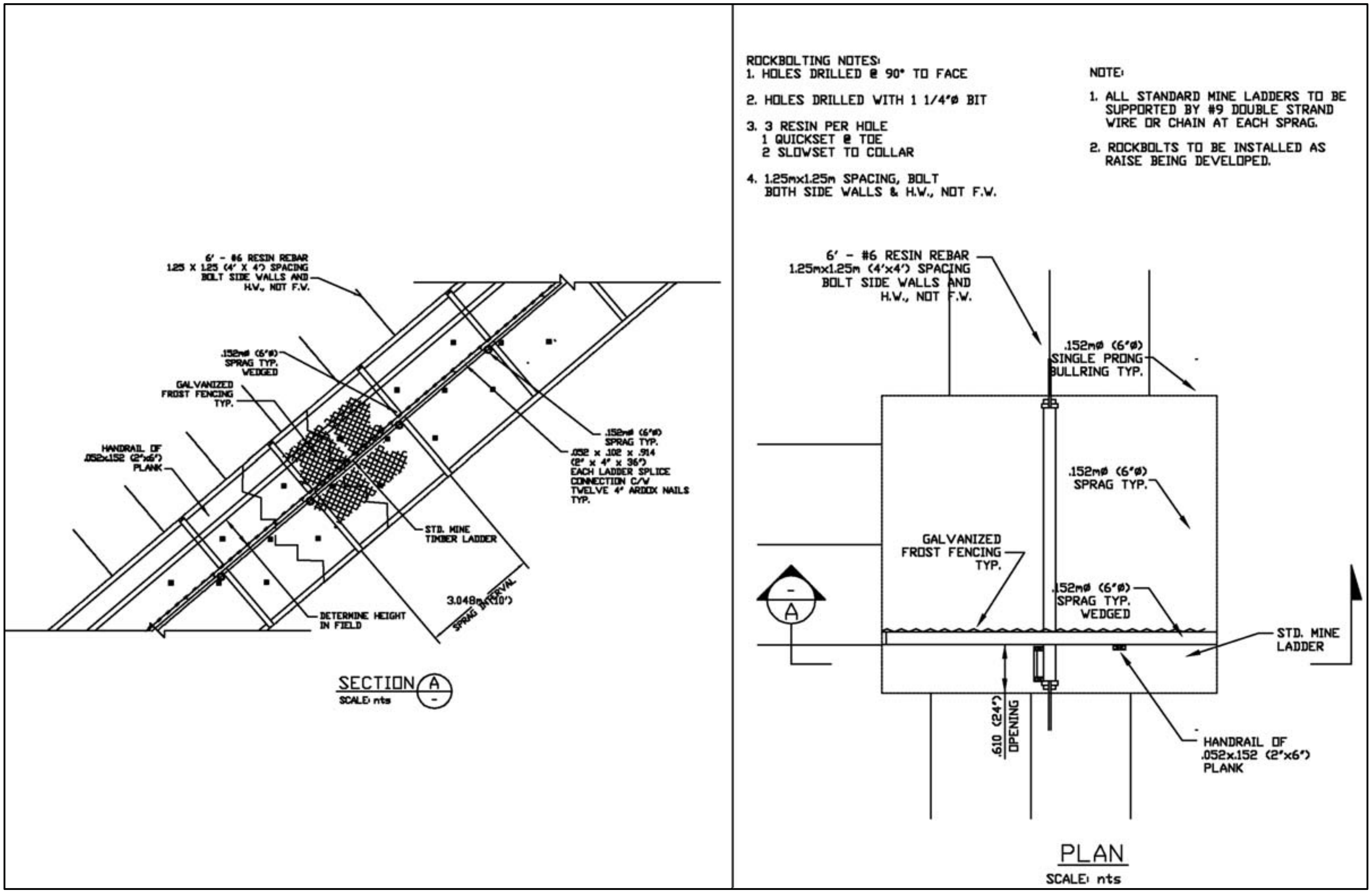
Category	Personnel
Subtotal Maintenance	15
Development Miner	12
Stope Miner	12
Scoop Operator	8
Truck Operator	8
Subtotal Mining	40
Total	**66**

16.2.11 Mine Ventilation

Primary ventilation will be provided through a downcast ventilation drift from surface. Fresh air will be forced down through the surface ventilation drift into the mine and through a series of ventilation raises into the stopes then will exhaust up the main ramp to surface. Ventilation raises will be equipped with a ladder to provide alternate egress from the mine (see Figure 16-15: Standard Vent-Escape RaiseFigure 16-15).

Two 200 kw fans each delivering 80 m^3/s (170,000 cfm) and a direct fired propane mine air heater will be installed on surface on the top of the ventilation raise. Fans will be designed to supply air into the mine for a total mine supply of 160 m^3/s (340,000 cfm). Ventilation doors and bulkheads will be constructed at strategic locations throughout the mine to prevent short-circuiting.

Figure 16-15: Standard Vent-Escape Raise



16.2.11.1 Ventilation Requirements

The mine ventilation system is designed to support the planned fleet of diesel haulage equipment as well as provide adequate ventilation for drilling and other activities. The provision of sufficient dilution air for the diesel haulage equipment is the dominant requirement.

The mine airflow estimates were estimated in two ways; one by stacking with the equipment being applied at 100%. The second was by applying the minimum requirement of 71 cfm/HP for all equipment. Both gave very similar results and the first was used as the basis for design after an allowance for leakage.

The fleet of diesel operated equipment planned for use in the underground mine with the related ventilation requirements is listed in Table 16-12.

Table 16-12: Mine Ventilation Requirements

Description	HP per Unit	No. of Units	Total HP	MSHA (cfm/unit)	Circuit Stacking (%)	Required Airflow Stacking	Required Airflow @ 71 cfm/HP
Truck 32 t	400	2	800	65,000	100	130,000	56,800
Scooptram 3.8 m^3 (5 yd^3)	193	3	579	25,000	100	75,000	41,109
Development Jumbo – 2 boom	160	2	320	14,000	100	28,000	22,720
Rockbolter	138	1	138	12,000	100	12,000	9,798
Scissor Lift	138	1	138	12,000	100	12,000	9,798
Anfo Loader	138	1	138	12,000	100	12,000	9,798
Jeep – Toyota	128	5	640	7,500	100	37,500	45,440
Sub Total			2,753			306,500	195,463
Allowance for leakage @ 10%						30,650	19,546
Total						337,150	215,009
Use for Design (cfm) (use the larger of the airflow calculations)						340,000	
Use for Design (m^3/s)						160	

16.2.11.2 Underground Ventilation System

Ventilation has been planned with an intake fresh air system consisting of a vent drift from surface connected to a series of interconnected raises and an access ramp connecting to each of the stope entrances. Fresh air will be drawn into the stopes via the access drift using fans and vent tubing and exhausted through the Cut-and-Fill stopes to the access ramps, which will exhaust to surface. Block 5 will be fed fresh air from the vent drift via steel vent ducting exhausting to surface via the main ramp.

The intake raise system for each of the mining zones has a similar design. The short ventilation raises are 2.7 m x 2.7 m with a manway installed to provide for secondary egress. The raises are generally inclined at 50° so that the manways can be installed continuous without landings. There is an individual short raise between each of the Cut-and-Fill stope access drifts extending down to the bottom of the mine.

Fresh air will be forced down each of the raises by a surface mounted fan and then down through the vent drift system. Ventilation bulkheads with fan manifolds and/or regulators and mandoors will be constructed at each Cut-and-Fill stope access to manage the air to be delivered to the various stopes. Generally, a fan and ducting will be used to move the air from the raise bulkhead into the stopes and to the working face. The ventilation air will generally exhaust through the stopes, then out the access drift to the ramp and to surface. The air will be provided by the surface fans with 160 m^3/s (340,000 cfm) of airflow through its own dedicated intake raise.

16.2.11.3 Development Ventilation

The ventilation for the first phase of development will be via 48" ventilation ducting. The main ventilation raises to surface will be developed at this point to serve as an intake and then the ventilation fan will be moved to the base of this raise. Based on air column calculations, the maximum volume that can be provided to this area using a 48" diameter vent with a 150 hp fan is 75,000 cfm. Upon completion of the ventilation raise to surface, the permanent fan and mine air heater will be installed. As the ramp is deepened, extension of the downcast raise will be given high priority so that the fans and vent ducting can be moved level to level as the ramp advances.

Cut-and-Fill stope development is supplied with fresh air via a series of fresh air raises, developed in the stope access cross cuts. Installation of 150 HP fans and 48" vent tubing will be used to deliver 35 m^3/s (75,000 cfm) to the sublevels. Cut-and-Fill drifting development will require 30 m^3/s (65,000 cfm).

16.2.11.4 Surface Ventilation Fans

Two 200KW fans will be installed on surface on the start of the ventilation drift to supply air to the mining zones for a total mine air flow of 160 m^3/s (340,000 cfm). Fans will be equipped with a direct fired propane mine air heater to prevent freezing during the winter months.

16.2.12 Underground Mechanical and Electrical Installations

16.2.12.1 Compressed Air

The compressed air requirements for underground mining are estimated at 0.7 m^3/s (1,500 cfm). Compressed air requirements were minimized by equipment choice. Compressed air is necessary for drilling, pumping, loading explosives and to ensure supply of air to refuge stations. At the start of development one 0.7 m^3/s (1,500 cfm) electric compressor will be used to supply compressed air needs. The compressor will be housed in a building close to the portal.

The compressed air will be distributed underground via a 150 mm (6") schedule 40 steel pipeline mounted in the main ramp.

16.2.12.2 Fresh Water

Fresh water for drilling and washdown will be distributed underground from the surface water supply system via a lightweight pipeline located in the main ramps. This pipeline will be 100 mm (4") diameter from surface to the main north-south junction of the ramp from which point separate 52 mm (2") pipelines will be installed.

16.2.12.3 Mine Water Discharge

Mine water will be discharged to surface from a series of sumps equipped with submersible electrical sump pumps. A preliminary estimate of the expected volume of water requiring discharge from the mine is approximately 0.946 m3/min (250 USGPM).

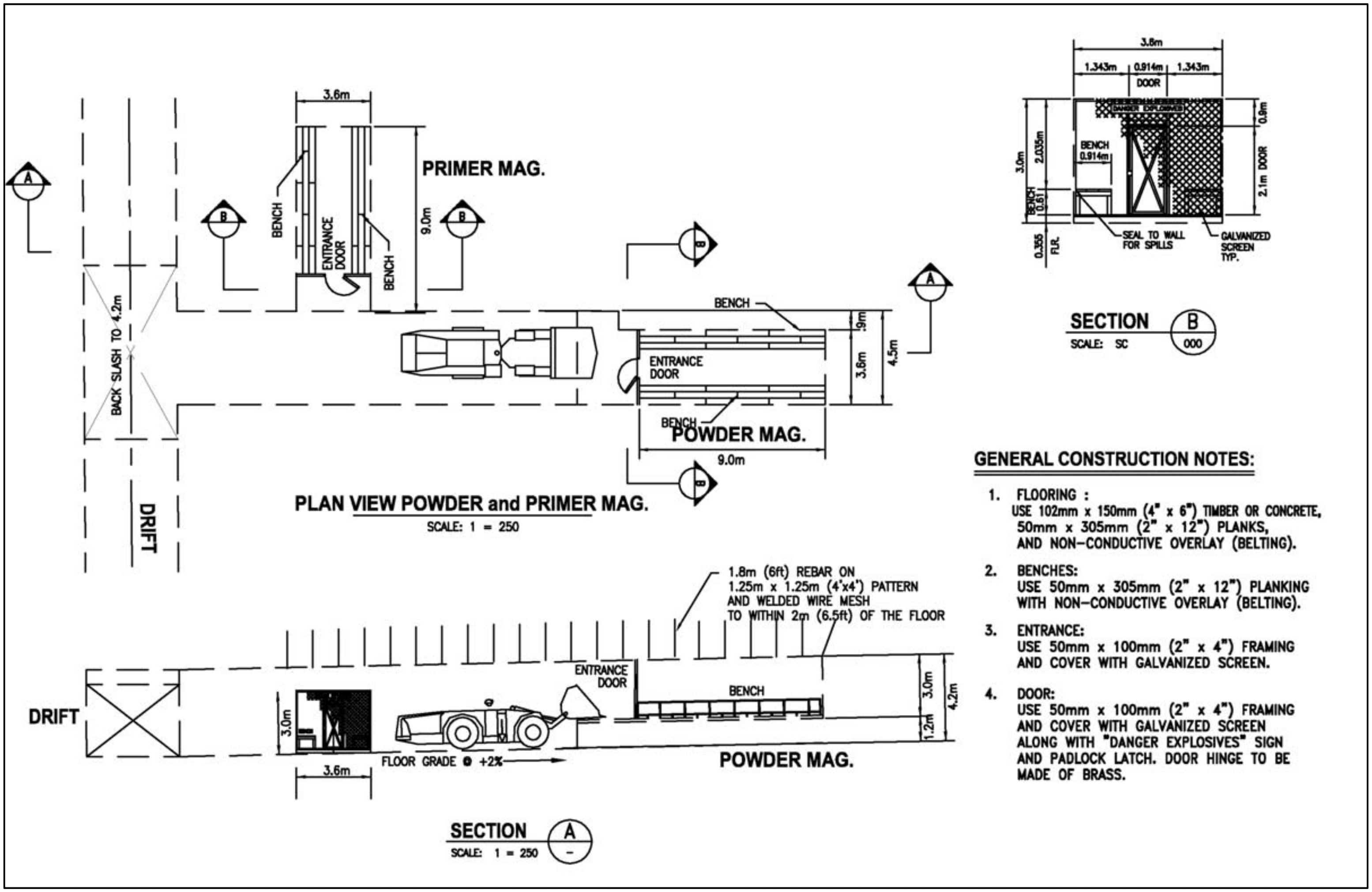
The regular water inflow during the ramp development will be handled initially by a stage pump system using 13 hp ,58 hp and 140 hp pumps. Main sumps will set up during ramp development to handle the maximum expected inflow of 0.946 m3/min. The main collection sump for the mining zones will be located on the main ramp collecting the water from all mining areas. A 6" discharge line will be installed in the main ramp from the clear water sump to surface. No allowance for significant inflows from the trench zone has been made in the above designs. Mitigation of inflows are addressed by managing the gap to the trench zone in the design approach for modified Cut-and-Fill.

16.2.12.4 Explosive Storage

Explosives will be brought to the site as required by a licensed contractor and stored underground. There will be no storage of explosives on surface, except in the earliest stages of the mine development. Explosive magazines will be licensed according to Mine Regulations. This study is based on the use of ANFO, however, emulsion explosives will be used where needed. Emulsion explosives are more expensive but have a lower concentration of ammonia compounds.

Typical plan and section layouts for explosive and primer storage are shown in Figure 16-16.

Figure 16-16: Primer and Explosive Magazines – Plan and Section Views



16.2.12.5 Underground Communications

The underground communications will be by telephone. Telephones will be installed on surface in the office/dry building, and underground in refuge stations.

16.2.12.6 Underground Electrical Power Distribution

Power will be fed underground at 4160 V from the existing main surface substation. The primary 4160 V feeders will be installed in the ramps and will feed portable 1,000KVA, 4160/600V transformers for use by underground equipment. Portable substations are included in the equipment list.

16.2.12.7 Men/Materials Handling

All movement of personnel and supplies in and out of the mine will be through the ramp system. Service equipment will include a scissor lift truck for installing equipment in the mine, as well as ground support, and five Toyotas for personnel and light supplies.

16.2.12.8 Fuel and Lubricant Storage

Diesel equipment will be fueled daily on surface from an approved diesel tank at an approved fueling station. Lubricants will be transported in bladders underground to an approved lube station. Lubricants, including hydraulic oils, will be stored in appropriate self-contained modules (Lube-cubes or Sat-stats). The lube storage bays will be equipped with a fire suppression system and fire resistant doors as per regulated.

16.2.12.9 Sanitary System

Sinks for hand washing, with heated mine water and chemical toilets will be available at each mine refuge stations.

16.2.13 Emergency Systems

16.2.13.1 Mine Emergency Response System

The ScoZinc Mine will have personnel trained on mine rescue present on site at all times. A mine rescue station will be maintained at the mine site and will contain emergency equipment to adequately supply two teams. An underground emergency mine warning system will be installed which will introduce ethyl mercaptan (sour gas) from pressurized cylinders into the mine ventilation intake. This warning system can be activated manually or by phone.

16.2.13.2 Secondary Egress

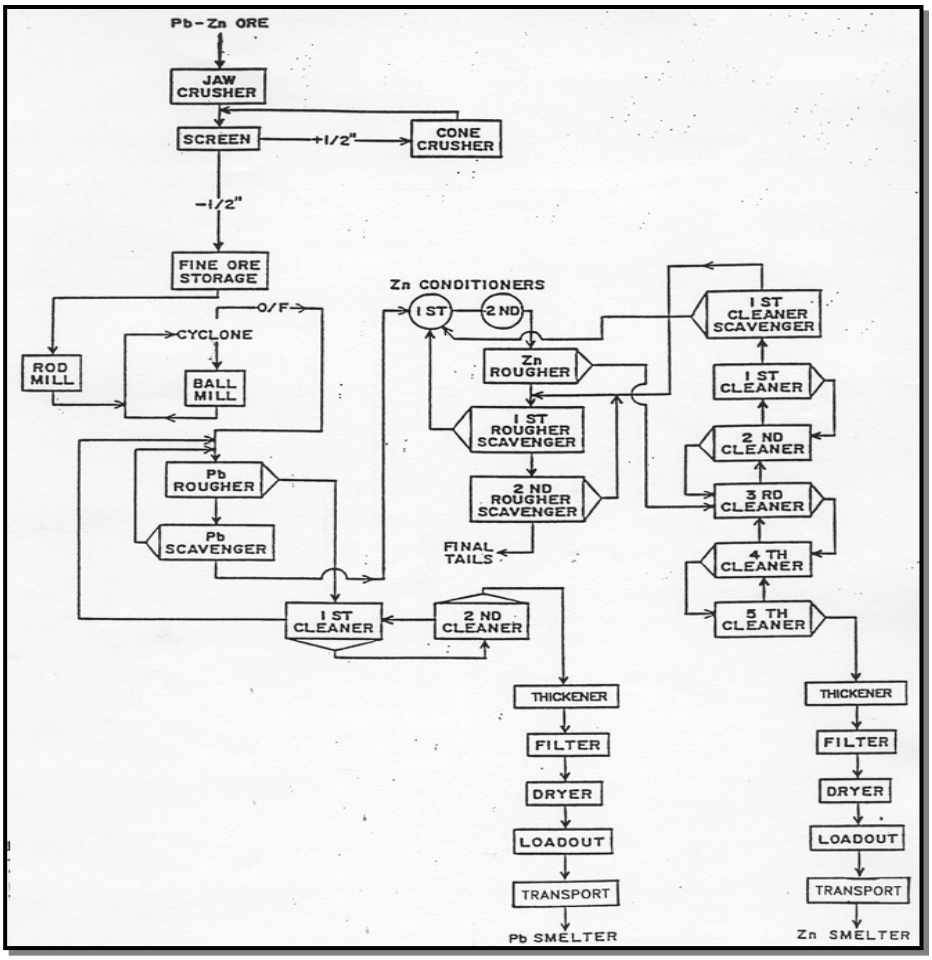
Secondary egress from the mine will be provided by the installation of ladderways in the ventilation raises. Each of these raise systems are located close to the main ramp and extend from the lowest level in that mining area through to a main vent raise that goes to surface. Each of the raise systems are fresh air intakes from surface so that they will normally always be assured to be in fresh air.

17.0 RECOVERY METHODS

17.1 Historic Flowsheet (pre-2008)

The existing plant operated for a significant time period prior to 2008 with an unusual flotation flowsheet. It incorporated a larger than normal number of flotation cleaning stages and significant recirculation of process flows. See Figure 17-1.

Figure 17-1: Process Flowsheet (pre-2008)



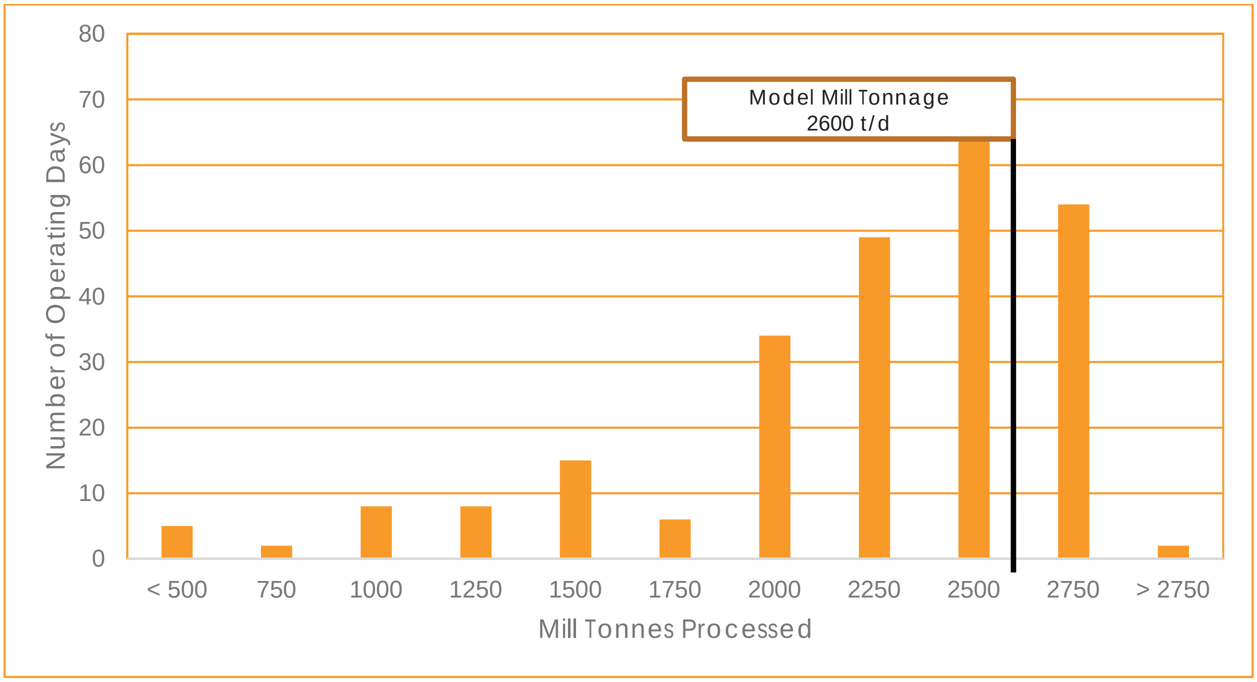
17.2 Plant Throughput and Challenges Identified Deficiencies

The plant throughput rate is based on operating data supplied by ScoZinc for the period of 30 November 2007 to 31 December 2008, which totals 271 operating days. During this period, the plant averaged 1,970 t/d with 85.3% availability. A total of 533,700 t were processed.

During this period, 17 days had zero production; excluding these dates makes the daily average throughput 2,100 t/d with an availability of 90.2%. Examining the zero production days, it appears that holidays—specifically Christmas and Thanksgiving, as well as a one week period between 16 and 21 August 2008—made up the bulk of the no production days.

A histogram of the throughput for the 2008 period is presented in Figure 17-2.

Figure 17-2: Daily Production Throughput (2008)



As can be seen from the production data presented in the histogram, during the 2008 operating period a total of 127 operating days had a production of greater than 2,500 t/d. The financial model relies on a daily average throughput of 2,600 t/d at 90% availability (328 days), which is 852,800 t per year (t/y) of mill throughput.

The spread of the data indicates that, although the majority of daily production throughput was above 2,000 t/d numerous days, a daily production throughput below 2,000 t/d will need to be addressed as part of the restart capital plan. The following deficiencies were identified as contributors to the poor plant reliability and performance.

17.2.1 Mining

- To achieve production targets, dewatering of the pit will be a critical success factor. In addition to dewatering capacity, establishing control of ingress, utilizing wells, and quality of the water being removed were concerns.
- Sufficient mill feed stockpiles will need to be established to support steady state production and allow for blending to better manage grade control.

17.2.2 Crushing and Grinding

- The size of run-of-mine resource varied dramatically, and with limited primary crushing circuit capacity, the crushing plant throughput was reduced or restricted. The crushing circuit consisted of a primary jaw located in the mill, with a single secondary cone crusher in closed circuit with a single deck screen.
- The ore entry point into the mill was open to the elements, resulting in wet feed and freezing conditions on a regular basis.
- The crushing circuit capacity was limited as a single deck screen 6’ × 12’ and a single secondary crusher struggled in terms of product sizing and overall capacity.
- The fine ore bin discharge used slot feeders that were neither controlled nor easy to maintain. This resulted in poor control of rod mill tonnage and challenges in both keeping the fine ore bin materials moving and keeping them properly maintained.
- There was insufficient process water available for rod and ball mills at higher throughputs. This limited production throughput and made grinding circuit control difficult.
- The grinding circuit was manually controlled by the operator.

17.2.2.1 Flotation

- There was no feedback to the operators on critical process streams assays. With fully manual and visual control, pH was controlled manually by the operator.
- All reagents were fed at a fixed rate based on the settings established by the operator.
- Flotation cell levels were controlled manually by the operator.

17.2.3 Dewatering and Filtration

- For both the Pb and Zn thickeners, the feed well did not achieve the design intent to evenly distribute feed, provide for de-aeration, and reduce turbulence, facilitating better solids settlement.

- The Zn thickener rakes were manually controlled, and with changes in thickener, loading, product density control and thickener functionality was compromised.
- The thickeners were manually operated, making discharge density control difficult.
- The disc filters and oil fired dryers were difficult to maintain and expensive to operate.

17.3 Proposed New Flow Sheets

The new proposed process flowsheets are shown in Figure 17-3, Figure 17-4, Figure 17-5, and Figure 17-6.

Figure 17-3: Proposed New Flow Sheet – Primary Crushing

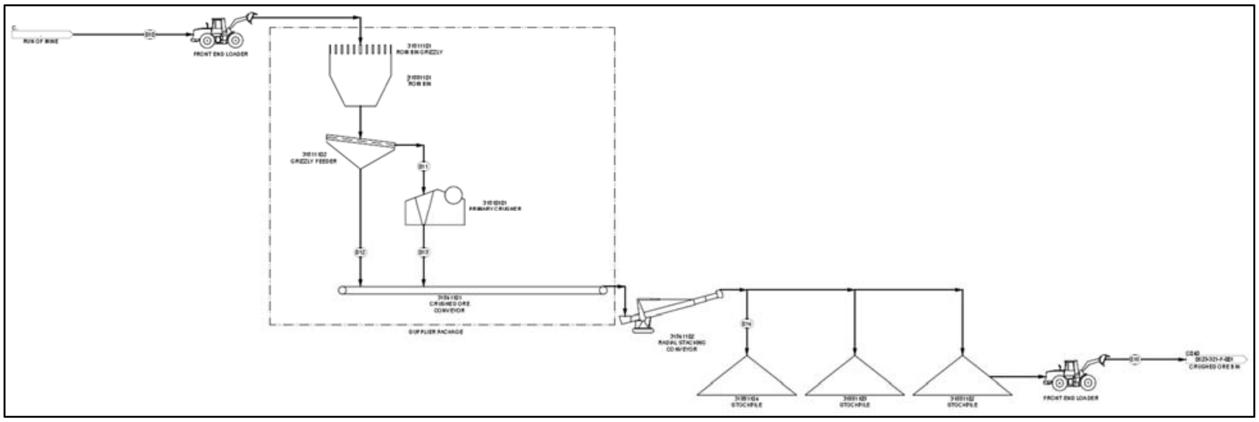


Figure 17-4: Proposed New Flow Sheet – Secondary Crushing

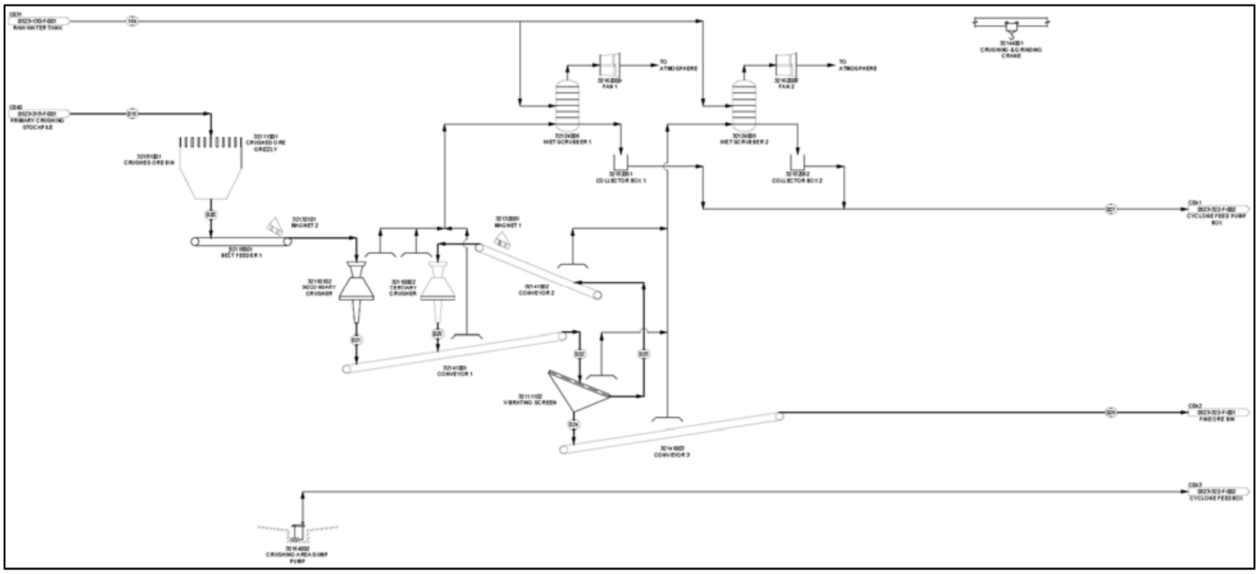


Figure 17-5: Zinc Flotation Process Flow

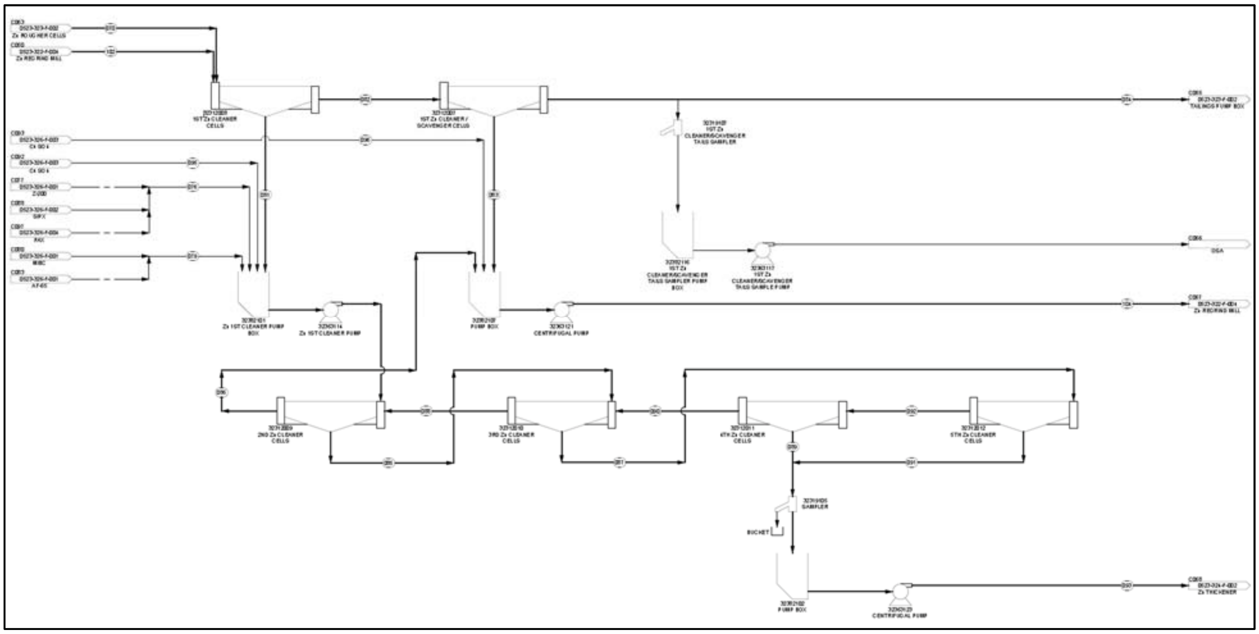
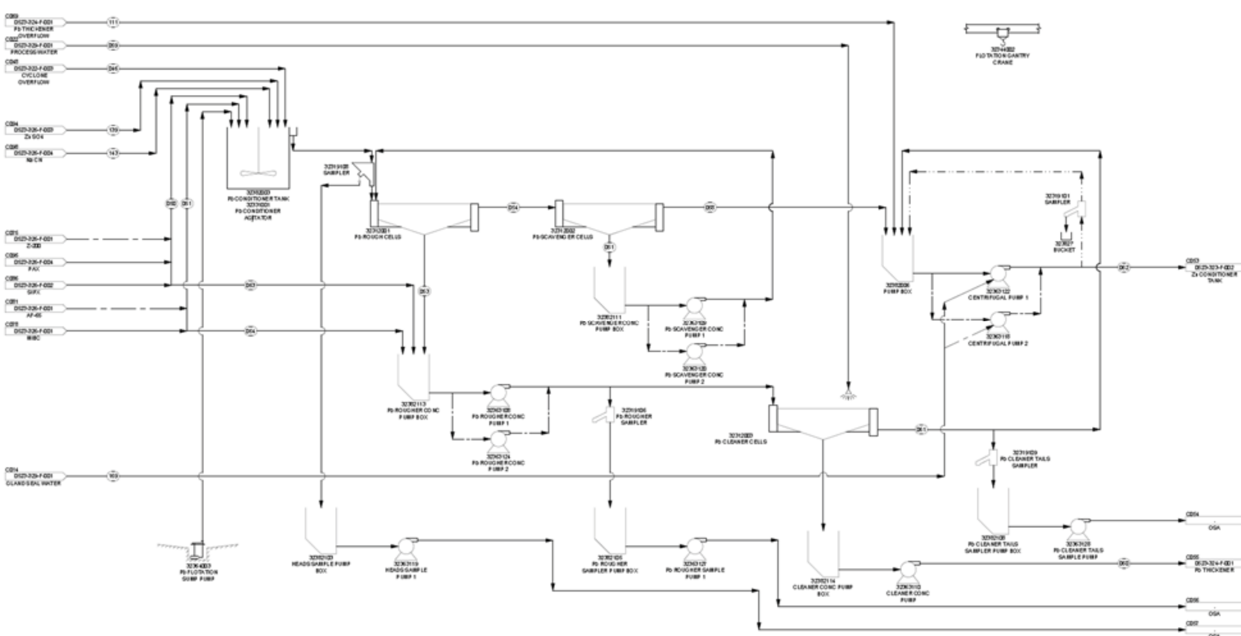


Figure 17-6: Lead Flotation Process Flow

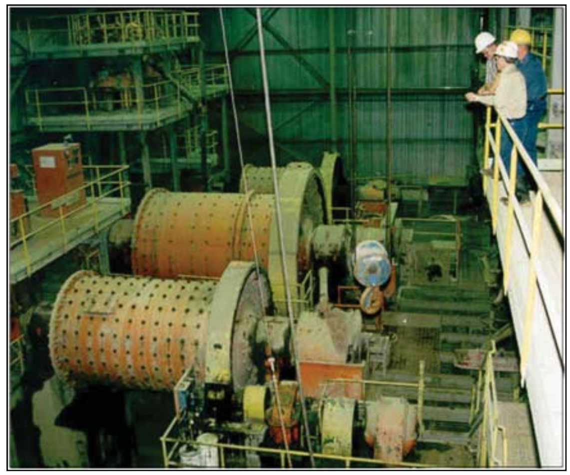


17.4 Proposed Plant

The ScoZinc processing plant was constructed during the late 1970s by Canada Wide Mines (Esso). Esso operated for less than two years, during 1979-1981. Seabright converted the mill to process gold during the mid-to-late 1980s. Westminer later reconverted and updated the mill to process Zn and Pb, then operated it for a short time during 1989-1991. In all, 1,795,271 t of Zn and Pb resource have been processed in the mill (see Table 6-1: Historical Milling Records).

External and internal views of the plant are shown in Figure 17-7.

Figure 17-7: Views of the Outside and Inside (Right) of the Mill



The mill building housed the primary jaw crusher, fine crushing, grinding, flotation, reagent storage and mixing facilities, and concentrate dewatering equipment. In addition, the mill offices, an analytical laboratory, and metallurgical laboratory are located in the building.

17.5 Mill Operations

Copies of the original Kilborn Engineering drawings are available, although plant design criteria could not be found. The nominal mill capacity was shown to be 1,500 short dry tons per day, or 1,360 dry metric tonne (dmt) per day. ScoZinc proposes operations at an average rate of 2,600 dmt per day.

To achieve the increased mill feed rates, plant modifications are proposed to mitigate most of the problems that challenged the previous operations. In addition, an effective preventive maintenance program is proposed to increase the plant availability from 85% to 90%; a more reasonable value for a plant of this type. A rigorous operating crew training program is planned to improve safety and operating efficiencies.

In order to provide a steadier supply of feed to the mill, the mine dewatering system will be upgraded based on historic experience to an average capacity of 5,900 gpm. This should reduce the number of mining disruptions due to water ingress into the pit. Where required, wells are to be used to better manage volume and quality of discharge.

The mine plan includes the buildup of a stockpile in front of the mill and installation of a primary crusher and radial stacker. This will provide the opportunity to establish a steady mill feed tonnage and improved grade control as blending is now possible. The primary jaw crusher will reduce the feed size to the crushing circuit which should improve overall crushing circuit performance and throughput.

Disruptions in mill throughput rates were principally attributed to a lack of capacity in the crushing plant, and difficulties caused by significant amounts of "sticky fines" in the mill feed. A temporary portable crusher was installed in May 2008 to provide short-term relief during a period of crusher maintenance. Mill throughput rates as high as 2,500 dmt per day were achieved once the supplemental crushing equipment was brought on line. The new process plan is to install new primary and secondary crushers with a refurbished tertiary crusher and an increase in the screening capacity to mitigate most of the materials handling problems.

The current single deck 6' × 12' screen is to be replaced with a double deck 6' × 20' screen with appropriate opening sizes on the decks.

The feed system from the fine ore bin has been identified as a key area for improvement and automation. The slot feeders are to be replaced to improve control and improve material movement. Plans are in place and monies have been budgeted to make these improvements. With the existing belt scale, and drive control on the fine ore bin feed system for the rod mill, feed rate will be automated to maximize feed rates and efficiencies.

An increase in primary grind size distribution is probable as the plant throughput is increased; no definitive data on the impact of increasing throughput is currently available. The planned crushing and grinding improvements are a proactive step in addressing the increased throughput required and to produce the desired flotation feed.

The grinding circuit comprises an 8' × 12', 400 hp rod mill and an 11' × 15', 900 hp ball mill. Rod and ball mill work indices were reported by SGS Lakefield (26 November 2007) to be 11.7 kWh/t and 10.9 kWh/t, respectively. With improved feed size control from the crushing circuit and grinding circuit, throughput and reliability should improve. In addressing the shortage of process water and introduction of automation in the grinding circuit throughput, liberation and reliability are anticipated to improve.

The metallurgical performance of the flotation circuit was reasonable in 2008, notwithstanding what appears to be excessive circulating loads due to the complex flowsheet selection. Inadequate flotation capacity generally results in high flotation pulp densities, inadequate retention times, excessive froth, and lip loadings; symptoms that are not readily evident, but individually and collectively adversely affect flotation performance.

The existing plant is virtually devoid of basic instrumentation and process control systems. Improvements in operations will be realized by the staged introduction of prioritized instrumentation and process control systems. To support the flotation circuit improvements, the introduction of the on-stream analyzer, and modernization and digital control of reagent addition systems, Ph control, and flotation cell level control are seen as critical elements in facilitating the improvements.

Promptly upon achieving grinding circuit stability, regular surveys of grinding and flotation circuit products will be performed. Based on the results of this work, modifications might be required to provide optimum metallurgical performance at the higher mill throughput rates.

The proposed improved concentrate thickeners are of adequate size to accommodate the planned increased plant throughput. The plan is to upgrade the feed wells to both thickeners to modern standards to improve performance. The Zn thickener rakes will be replaced and automated to better manage the thickeners output and performance. The discharge pumps for both thickeners will be reviewed and, if required, density control and automation will be enhanced.

The vacuum filters were inspected and found to be in very poor condition. ScoZinc has removed the vacuum filters and dryers, and will replace them with two pressure filters which will provide increased flexibility in the concentrate dewatering circuit. Elimination of the oil used to fuel the two concentrate dryers will offset the capital cost of the new pressure filters.

An estimation of the expected metallurgical performance of the modified grinding / flotation plant is shown in Table 17-1 and is used in the financial model.

Table 17-1: Expected Metallurgical Performance of ScoZinc Operational Plant

		Total Y1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Total LOM
Tonnes	t	**568,341**	851,939	852,816	852,816	852,816	852,816	867,319	852,816	6,551,679
Feed Grade	% Pb	0.67	1.28	1.80	1.66	1.48	2.18	2.07	1.07	1.57
	% Zn	2.51	3.00	2.98	3.45	3.17	3.55	4.04	1.62	3.07
Lead Concentrate										
Lead Recovery	%	85.7	85.7	85.70	85.7	85.7	85.7	85.7	85.7	85.7
Lead Con Grade	% Pb	71.0	71.0	71.0	71.0	71.0	71.0	71.0	71.0	71.0
Lead Conc. Pb Content	t	3,267	9,370	13,163	12,100	10,851	15,933	15,372	7,820	87,876
Zinc Concentrate										
Zinc Recovery		86	86	86	86	86	86	86	86	86
	% Zn	57.0	57.0	57.0	57.0	57.0	57.0	57.0	57.0	57.0
Zinc Conc. Zn Content	t	12,287	21,973	21,824	25,268	23,228	26.007	30,121	11,881	172,589

18.0 PROJECT INFRASTRUCTURE

Infrastructure requirements for the ScoZinc open pit mine are discussed within this section. Halifax is the provincial capital of Nova Scotia and, in combination with surrounding communities, forms a major center of population, government, business, education, industry, and transportation services. The mine site is 55 km northeast of Halifax and is directly accessible from the paved provincial Highway 277 or 224.

18.1 Transportation and Offsite Infrastructure

Zn concentrate was trucked in bulk to Sheet Harbour, Nova Scotia, where it was loaded onto a bulk ocean carrier. Pb concentrate was loaded into lined ocean shipping containers, and trucked to the Port of Halifax.

Robert Stanfield International Airport is located approximately 20 km southwest of the property and provides both daily domestic and international airline services.

The property area is rural and has been extensively developed for agricultural purposes in the past. Access to mainline rail facilities is possible at the nearby town of Milford (8 km by paved road) and direct access to deep-water shipping facilities with post-Panamax capacity is present through the ice-free, deep water port of Halifax (Figure 18-1).

Figure 18-1: Port of Halifax

Year round, deep water access, storage and ship loading facilities for Pb and Zn concentrates are also available at the seaport of Sheet Harbour, 80 km from the mine site over paved roads. Sheet Harbour is a natural harbour on the Atlantic coast that remains ice-free in the winter months. The Harbour can handle vessels up to 40,000 t in displacement (Figure 18-2).

Figure 18-2: Sheet Harbour

The building in which the concentrate is stored is in good shape. Within it is an area where the concentrate is placed onto a conveyor. There are another five conveyors that are placed daisy-chain fashion to transfer the material to the docked vessel. Those five conveyors are being stored outside close to the building and will require some minor maintenance prior to start-up.

Rail transport facilities have also been used for concentrate shipping. A railway siding is located in Milford, eight kilometers from the site.

18.2 Onsite Infrastructure

Due to the mine's operational history, existing onsite infrastructure will continue to be maintained and used as the ScoZinc mine goes into production.

The required infrastructure for the current Main Zone is in place. The runoff from the existing waste pile has a natural path to the tailings pond on the east side, but there will be some contouring / landscaping required for the west side of the pile to ensure a directed flow into the tailings pond. As the size of the waste rock pile increases to the east and the south, contouring will have to continue on both the east and the south side of the pile. Some minor road development will be required during the pre-strip to access the north waste pile and the expansion of the south waste pile. The new north waste pile will require contouring at the pile base to ensure that the drainage from the pile is directed back into the pit. In addition, the Northeast Zone will require service and haul roads and other minor infrastructure, such as out-buildings, staging areas, and working areas. As this is a PEA, the detailed design of these improvements has not been performed.

The existing roads are in adequate condition and will require minor realignments, extensions, intersections, and signage to accommodate the increased traffic and additional operational areas.

The main ScoZinc Access Bridge was inspected by Allnorth on 27 July 2011. The assessment indicated that additional to regular inspection and maintenance, there are signs of distress and deterioration that require replacement and/or repair within 2-3 years. With the use of large vehicles, such as 777 haul trucks, the bridge will have to be upgraded prior to commencement of production.

Power is supplied through a 25 kV line fed in from the Emsdale sub-station. The line crosses the Gays River on the northwest side of the property and traverses a minor road towards the pit. In this immediate area there is a pole mounted meter. A new pole line will continue from that meter in the northerly direction so that the future expansion of the pit is skirted. It will then proceed along the river and follow the main access road back to the yard transformer. The current section of the pole line from the meter to the yard transformer will be removed.

Most of the mill's water requirements are satisfied by in-process recycling. However, make-up water is drawn in from the Gays River by use of a 15 hp, 320 gpm pump located near the main road bridge at the highway. The water from the pump is brought into the site via a pipeline that runs along the main access roadway. The system is adequate for future production.

The process or reclaim water is currently supplied to the mill from the pumphouse at the finishing pond. It has a 700 gpm pump and an 8" supply line to the mill. At peak production, the mill was running out of water, so this system will have to be upgraded by putting in a larger pump and an additional pipeline. The existing tailings pond is large enough for the life of the proposed operation. It is located just south of the mill on the footwall side of the deposit. The pond's design capacity was ten million tonnes. Approximately two million tonnes of tailings have been stored there, indicating a current capacity of about eight million tonnes. One raise, of one meter, is planned for the tailings dams and is accounted for in the economic analysis. The decant from the tailings pond is directed into a finishing pond and from there is discharged into the Gays River. The finishing pond is also suited for the future production.

The water from the dewatering of the pit also flows into the tailings pond. There are currently five dewatering wells, located on the east side of the pit. The pumps that are being used will have to be replaced or rebuilt and two new pumps will have to be purchased. The 18" line that carries the water to the tailings pond loops around the north end of the pit. It will have to be moved and lengthened to ensure that it is further north of the pit expansion.

There is sufficient area for overburden storage on the property.

The diesel-powered fire pump and the water storage tank are in good shape and will only need routine maintenance prior to being ready for production.

The storage areas are being well maintained and are suitable for start-up. However, some of the administration facilities have some mould and interior work will be required to rectify that hazard.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Markets

There are several potential markets for concentrate sales. Historically (2007-2009), the ScoZinc concentrate was sold to smelters in Europe, South Africa, and Asia through contracts with major trading companies. The ScoZinc mine concentrates are deemed highly desirable by smelters due to their high concentrate quality, grading 57% Zn and 71% Pb, and low levels of deleterious metals. These characteristics enable marketing of the ScoZinc concentrate at favorable terms, as its purity is suitable for blending by smelters worldwide.

19.2 Concentrate Sales

Historically, ScoZinc established multi-year concentrate purchase contacts with MRI Trading AG (MRI) and Trafigura AG (Trafigura) under terms consistent with the market terms at that time. The purchase contracts accounted for 100% of Zn production and 100% of Pb production in which Trafigura and MRI each had the obligation to purchase 50% of the produced Zn concentrate. Trafigura also had the obligation to purchase 100% of the Pb concentrate produced. In both cases, ScoZinc had obligation to sell its Zn and Pb concentrate production under the established quantities and terms.

ScoZinc expects to establish concentrate purchase contracts once again with one or more metal trading companies under terms consistent with the current market terms. ScoZinc is engaged in negotiations with a number of off-takers, but has not yet finalized an arrangement.

This PEA assumes that long-term treatment and refining charges will escalate as shown in the table below.

Table 19-1: Treatment and Refining Charges

Commodity	Year 1	Year 2	Year 3	Rest of Mine Life
Zinc ($ per DMT)	$70	$100	$150	$190
Lead ($ per DMT)	$70	$95	$135	$150

19.3 Zinc Analysis

For the purposes of this PEA update, no new information has been used and the text has remained consistent with what was used in the 2013 update.

Based on the previous work and analysis as supplied by ScoZinc:

> *"Wood Mackenzie was engaged in early 2013 to provide forecast data on zinc and lead treatment and refining charges for zinc and lead concentrates. This section and section 19.4 stem from their analysis. Annual global treatment and refining charge and metal price forecasts for zinc and lead are presented in Figure 19-1. The estimates are based on customary quality of concentrate and may vary with quality outside of the normal range."*

19.3.1 Zinc Consumption

The near term outlook for Zn consumption is positive, with demand expected to grow at 5.0% per annum until 2015 (Wood Mackenzie, January 2013). This growth rate largely reflects continued industrialization in China, where forecast capital investment plans support steel, and hence galvanizing, demand.

In the longer term, China's economic expansion will become less Zn-intensive as the authorities move to gradually restructure the Chinese economy. This will be done by reducing its dependency on capital investment and exports and by increasing the contribution to growth from the less metals-intensive domestic service sector. This reduction in demand will not be compensated by equivalent growth elsewhere in the world, and as a consequence, global Zn consumption growth will moderate in the long term to average 3.4% per annum. over the period 2015-2025.

Despite the slower pace of growth projected for Zn demand in the coming years, the urbanization and industrialization of China and other economies will ensure that incremental growth in global Zn consumption remains significant in absolute terms, with forecast growth averaging 570k t/a over the period 2015-2025.

19.3.2 Zinc Mine Supply

In the near term, the outlook for mined Zn supply is one of sustained, albeit limited, growth, with mine capability projected to increase to 14.7M t/a Zn by 2015 at a compound annual growth rate (CAGR) of 1.1%. Forecast mine production capability is based on the currently identified mineral resources of individual mines. As such, the depletion by mining results in a declining trend of mine production capability, which is projected to be 12.1M t/a in 2020 and 9.8M t/a in 2025. New capacity to meet the growth in demand for Zn will arise from expansions, mine life extensions at existing mines, the development of new capacity from projects that are currently being advanced through the development process, as well as new discoveries. Mine production is projected to increase to 14.9M t/a in 2015 and then to grow at a CAGR of 3.3% to 20.9M t/a by 2025, setting an annual average requirement for new output of 600k t/a.

19.3.3 Market Balance and Prices

19.3.3.1 Concentrate Market and Treatment Charge

In 2012, there was serious underperformance of Chinese smelters with global refined output contracting year-on-year. Global mine output continued to grow, and the concentrate market moved into surplus. Annual surpluses are expected to persist to 2016, before the market moves to deficit with concentrate inventory trending down and stabilizing at around 40 days of smelter demand.

The Zn concentrate treatment charge (TC) is determined by bilateral negotiation between mines and smelters. It is positively correlated to changes in the Zn price and inversely correlated to the supply of concentrate, with restricted supply resulting in reduced TCs and surplus supply resulting in higher TCs.

Our forecast realized treatment charge ranges between $240/t and $430/t concentrate over the period to 2020 depending on the projected Zn price and concentrate stock availability in any year. The increase and range in treatment charges reflect the price participation component of smelters as metal prices rise with forecast supply shortfall.

19.3.3.2 Refined Balance and Zinc Price

The normal inverse correlation between refined inventory and the metal price (increasing stocks and falling price) was reversed following the 2008/09 recession, with both stock and price moving higher due to investment fund activity in the base metals asset class. Exchange stocks on the LME and SHFE exceeded 1.3M t in 2012, equivalent to 36 days of demand, and total implied inventory was 102 days of demand. Following 4 years of surplus, we judge that China moved to a refined deficit in 2012, but with the rest of the world still in modest surplus. Over the medium term, we project a succession of annual market deficits that will return global implied refined inventory levels to a historic norm of between 50 days and 60 days of consumption. This fundamentally-sound outlook, further supported by our analysis of project incentive pricing and the operating cost/price relationship of the global Zn mining industry, underlies our view that the Zn price will increase from current levels of around $2,000/t to a projected base case longterm price of $2,600/t in real terms (Wood Mackenzie, January 2013).

19.4 Lead Analysis

19.4.1 Lead Consumption

The near-term outlook for global Pb demand growth is positive, with demand expected to rise at 4.5% p.a. out until 2015. Although Europe remains largely at the mercy of prevailing economic conditions, US demand has recovered well and will continue to benefit from strong auto output and industrial battery sector growth. Chinese growth, whilst slowing, is still robust, driven by strong e-bike and auto markets and demand for stationary batteries in telecoms upgrades.

Longer term, global Pb demand growth is forecast to be 3.4% p.a. in the period 2016 to 2025. Although many growth opportunities still exist via further telecoms upgrades, Uninterruptible Power Supply (UPS) applications, stop-start battery technology, and growing global vehicle populations, growth will be partly offset by improving battery quality. Chinese infrastructure spending growth is also expected to slow, and although India and Brazil have good prospects, these are likely to unfold gradually over time (Wood Mackenzie, January 2013).

19.4.2 Supply

19.4.2.1 Lead Mine Supply

Global Pb mine capability is forecast to grow by a CAGR of 4.7% out until 2015. Most of the growth centres on China, but there is also a significant contribution expected from Latin America. This more than offsets a number of upcoming mine closures on reserve depletion, such as Brunswick in Canada and Kassandra in Greece.

Mine capability will decline post-2015, and it falls to probable and possible projects to provide new sources of mine supply. Taking these into account, global Pb mine production growth is forecast at 4.4% p.a. to 2025, after accounting for a general disruption allowance (Wood Mackenzie, January 2013).

19.4.2.2 Refined Lead Supply

Global refined production capability is expected to grow at a CAGR of 1.7% p.a. between 2011-2025. China will continue to drive growth, at 3.0% p.a. or 2.2M t, driven by new operations and capacity expansions at existing plants (Wood Mackenzie, January 2013).

19.4.3 Market Balance and Prices

19.4.3.1 Concentrate Market and Treatment Charge

Over the past decade, primary smelting capability has increased rapidly, mainly driven by expansions in China, resulting in a 10.5% p.a. increase in capacity, from 1.1M t in 2001 to 3.2M t in 2011. The move to Zn only or Zn-copper mine projects, coupled with robust Pb demand, has resulted in limited availability of clean, low silver concentrate.

The restricted supply of concentrate is set to prevail to 2018, as growth in primary output continues to outweigh growth in global mine supply. Concentrate stocks will remain at or near a low of 30 days of requirement. Scrap feed to primary smelters will be constrained by a tight scrap market and increasing competition from secondary producers. Beyond 2018, additional mine supply from new projects should increase concentrate availability as mine supply from new projects come on stream.

The Pb concentrate TC is determined by bi-lateral negotiation between mines and smelters. It is positively correlated to changes in the Pb price and inversely correlated to the supply of concentrate with restricted supply resulting in reduced TCs and surplus supply resulting in higher TCs.

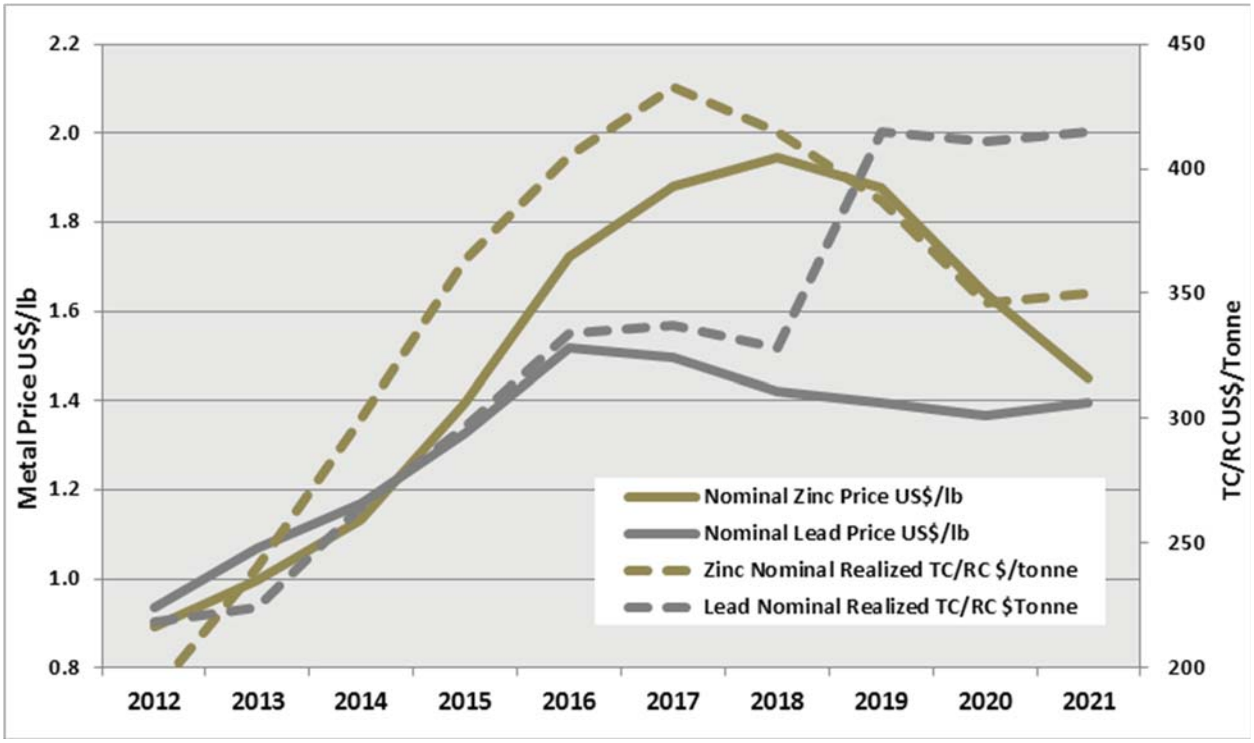
Our forecast realized treatment charge ranges between $224/t and $414/t concentrate over the period to 2020, depending on the projected Pb price and concentrate stock availability in any year. For the longer term, the projected base case average annual TC is $338/t concentrate in real terms specific to our long-term Zn price forecast of $2,500/t Pb in real terms (Wood Mackenzie, January 2013).

19.4.3.2 Refined Balance and Lead Price

After several years of surpluses, the refined Pb market returned to deficit in 2012 and is expected to remain undersupplied for the next few years. A period of robust global demand growth, coupled with tight raw material supply in both the primary and secondary (scrap) sectors will limit supply growth. The supply side is at further risk from increasing environmental legislation. Refined stocks accumulated over the past few years of surpluses will swiftly be eroded and stocks will bottom-out close to historical low levels. Prices will rise accordingly over the period, reaching a cyclical peak of $3,000/t in 2016.

This period of higher prices is expected to encourage more refined production on stream, and so from 2017, the refined market is forecast to return to surplus. Pb demand growth will stabilize following the post-recession recovery, and from 2015, our demand forecast reverts to a trend average (see Figure 19-1). From 2020, we have set our refined stocks in days of consumption at an average of 38 days, and our Pb price forecast reverts to a trend average of $2,500/t.

Figure 19-1: Wood Mackenzie 2012-2021 Zinc and Lead Price Forecast



19.5 Contracts

At the time this report is published, ScoZinc has not yet entered into any contracts as it relates to mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements.

As a basis for capital and operating costs updates numerous quotations have been received and used, but no contracts are in place for any of the goods or services.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

It is important to recognize that the ScoZinc Mine is an existing operation with substantial environmental databases, operating history, and valid permits and licenses that allow for the mining, processing of resources, and the shipping of concentrates. Roughly half of the resources used in this economic analysis are already under permit and mining of those resources (Southwest Expansion) can begin immediately.

Another important aspect of the project status with respect to permits, environment and community is that regulators and the community have experience with the project and environmental baseline conditions are already understood. In combination, these factors limit the overall permitting risk and anticipated timelines for permitting project expansions to include the entire mineral resource used in this analysis.

In addition, the risks and potential costs associated with environmental and community issues are well understood and based on operating experience and history of the mine. As such, the financials for environment and community matters that are input to the economic model are accurate to a feasibility level.

20.1 Environment and Existing Socio-Economic Conditions

20.1.1 Environment

The climate conditions for the Project area are based on the nearest climate station with historical data. The Upper Stewiacke climate station (operated by the Meteorological Service of Canada; Government of Canada 2017) is located approximately 35 km northeast of Gays River at a similar elevation. Based on data collected between 1981 to 2010, the average total annual precipitation is 1,363.6 mm (water equivalent), which includes 248.1 cm of average snowfall per year and 1,115.5 mm of average rainfall per year. Monthly precipitation amounts remain fairly constant throughout the months of May to August increasing through September through December. Average annual temperature is 6.2 °C, with an average monthly range from -6.8 °C in January to 18.4 °C in July.

The physical geography of the area surrounding the Project has been previously described (CRA 2011). Topography in this region of Nova Scotia is dominated by mainly Carboniferous rocks (shale, limestone, sandstone, gypsum) upon which deep soils derived mainly from glacial outwash have developed (Roland 1982 in CRA 2011). These central lowlands provide a topography that is variable in nature from lowland plains to rolling hills and rarely exceed 90 m above sea level (CRA 2011).

The local geology consists of a dominance of Lower Carboniferous (Mississippian Age) Windsor Group strata with occurrences of the Meguma mapped southwest and northeast of the Gays River/Cooks Brook area.

The Project area is well drained, with fine textured soil on hummocky terrain that lies in the southern extent of the Central Lowlands, adjacent to the Rawdon/Wittenburg Hills and the Eastern Interior ecodistricts (Neily et al., 2005).

The hydrogeological regime in the Project area is complex; controlled by a karsted gypsum / carbonate contact which has been in-filled with Cretaceous-age sands and clays. Two overlying Pleistocene glacial cycles and recent deposition of the river alluvium adjacent to the meandering Gays River complicate the hydrogeology. Several sand units form aquifers that are separated by zones of permeable clays which are probably interconnected in the karsted gypsum deposits overlapping the mineral deposit (CRA 2011).

The Gays River is the principal watercourse in the area, with its headwaters in Lake Egmont. The Main Branch of the Gays River flows north and west past the Project site, where it converges with the South Branch Gays River. Drainage for the Gays River sub-system of the Stewiacke-Shubenacadie River system collects from the valley sides to the north and south of the emerging Cooks Brook and Gays River (CRA 2011).

In the Project area, the Gays River is a meandering channel with overall low gradient and limited riffles and abandoned pools. The substrate sediments are predominately silt with very minor boulders and cobbles. The active channel averages 10 metres in width with a range of water depths from several centimeters to several metres (CRA 2011).

There are several wetland complexes in the Project area. In Nova Scotia wetlands are protected under the provincial *Environment Act* and an approval is required for their alteration. A wetland survey and compensation plan may be required as a component of the Project development for areas not currently under Industrial Authorization permit. Surveys of plants and animals have previously been completed for the general project area. Previous work has identified *Hepatica nobilus* (Round-Lobed Hepatica), which is considered endangered in Nova Scotia by the Nova Scotia Department of Natural Resources (NSDNR); the project Environmental Assessment approval requires a monitoring program for this species and it is expected that this would continue with the Project. Other flora species of interest include two plant species and one lichen species. Canada Lily (*Lilium canadense*), Canada Wood Nettle (*Laportea canadensis*) and the lichen *Sticta fuliginosa* are considered sensitive to human interaction by NSDNR.

No fauna species of concern have been identified to date within the site footprint. However, Wood Turtles (*Glyptemys insculpta*) have been observed within the local area. This species is categorized as threatened under both provincial and federal legislation. The project Environmental Assessment approval mandates the undertaking of an inventory and ongoing monitoring program for this species, and it is expected that this would continue with the Project.

Additional wildlife surveys will be required prior to disturbance in the Southwest Expansion area, as well as for the new development areas contemplated by this report.

20.1.2 Socio-Economic Setting

The Project is located in Cooks Brook, a small unincorporated community in the Halifax Regional Municipality (HRM) that borders the community of Gays River, Colchester County. This community lies between the larger communities of Middle Musquodoboit, Lantz and Shubenacadie. The population of the surrounding area is described by Nova Scotia Finance, Community Counts to fall within three "communities" namely Middle Musquodoboit, Lantz, and Wittenburg. The total population of these three areas is 6816 (2006 Census; community-specific information is unavailable from the 2016 census). About 28% of the population is under 20 years of age and 13% is 65 years of age or older. Population growth between 1996 and 2006 was about 3%. English is spoken by over 99% of the population. The average family income for the three communities in the area ranges from \$56,500 to \$67,000 per annum (the more affluent area being Lantz) (CRA 2011).

In the local area there is a range of land uses focused on resource based industries such as agriculture, forestry and mining. The mine site is in an agricultural area that extends from the Musquodoboit Valley north into Colchester County. Agricultural land use accounts for approximately 5% of the Gays River area.

The local area is primarily forested with mixed use (mainly residential and small business) located along the secondary roads. Sawmills and a wood pellet manufacturing plant are located near Middle Musquodoboit. Forested lands are primarily privately owned. Private woodlot owners are a significant source of supply to these facilities. ScoZinc owns about 50% of the property in the Gays River area (Figure 4-4).

Project permits currently require and mandate a Community Liaison Committee (CLC) composed of local residents and mine site staff. During mine operation, the committee meets on a regular basis to discuss issues from the community in regards to the mining operations. Past meetings have highlighted noise from the operations as a key issue for local residents as well as concerns about dust and potential impacts to Gays River quality and flow. Employment and opportunities for local people is also an important issue. The operation expects to draw substantial numbers of its workforce from the nearby communities and as such employees will also provide a valuable source of communication on local issues. It is anticipated that the requirement for a CLC will continue to be a requirement of Industrial Authorizations for future project expansions.

Local residents rely on wells for water supply. Potential impacts to groundwater wells from localized dewatering of the aquifers through mining or changes to aquifers from blasting shocks have been identified as a possibility. Bonding of $147,500 is in place with the Government of Nova Scotia for the purposes of supplementing water supply of local residents in the event of impacts to water supply wells. The bonding amount is based on residents within a certain proximity of the mine site. As the Project expands, the requirement for bond will likely need to be increased to account for the greater number of residents that fall within that perimeter.

Archaeological studies of the Project area have been undertaken for various phases of the project development. Given the proximity to Gays River and the long post-colonial history of the area, several, mostly minor, pre-contact and post-contact archaeological sites have been identified within the Project footprint. However, these previously identified sites have been studied, catalogued, and otherwise dealt with so that they require no additional action and will not inhibit mine development. Archaeological surveys of additional areas that may be disturbed as part of this Project may be required during the permitting process.

20.1.3 Summary of Environment and Socio-Economic Issues

Environmental studies of the project area have identified several plant and animal species of interest, including two that require an ongoing monitoring program. An additional wildlife survey will be required prior to disturbance in the Southwest Expansion area, and footprint-specific surveys may be required for new areas of disturbance anticipated by the development scenario identified in this report.

Community communication and involvement is mandated to a CLC through existing Project permits. This committee is and will continue to be the primary tool for communicating with local residents regarding their concerns or issues with the Project. Noise, dust, and impacts to water wells have been identified as key community issues. Bonding requirements have been established as mitigation for impacts to water wells and it can be expected that the overall bond requirement will increase as the footprint of the project increases and the number of residences within the certain proximity of the Project increases.

Archaeological sites in the area of the Project are not uncommon, and several minor sites have previously been identified within the project footprint; however, these sites have been assessed and will pose no obstacle to mine operation or development. Footprint-specific surveys will be required for new development areas and there is a possibility of identifying pre and post contact archaeological remains in the project area. Management of any such finds may require avoidance through adjustments in project plans or, if this is not possible, excavation of any identified sites prior to project disturbance.

20.1.4 First Nations

First Nation involvement with past operators of the mine and mill was meaningful. First Nations were involved in Mi'kmaq ecological knowledge gathering in the late 1990s, 2005, and again in 2012. In 2006, an archaeological site of significance (the Sinkhole Site) was mitigated using First Nations involvement and staff in an area of Gays River. The site had been planned for disturbance by a previous mine plan. Contact has been made with representatives from the closest First Nations community of Sipekne'katik First Nation and preliminary discussions held about mutually beneficial programs. ScoZinc will pro-actively engage in further discussions and are cognizant of the "Mi'kmaq - Nova Scotia - Canada consultation Terms of Reference".

20.2 Regulatory Process and Project Permitting

20.2.1 Mine Permitting – Nova Scotia

The Province of Nova Scotia has a well-defined mine permitting process. The pre-development permitting process can be generalized into two stages, defined as Stage 1 or the Environmental Assessment (EA) stage and Stage 2 or Permits, Leases, and Approvals. Stage 1 is completed first, followed immediately by Stage 2, and once Stage 2 approvals are in place, mining activities can commence.

20.2.2 Stage 1 – Environmental Assessment

Provincial

The EA for a mine proposal normally occurs following advanced exploration, and a positive economic analysis to warrant mine development. The environmental assessment process in Nova Scotia is regulated under the provincial *Environment Act* and *Environmental Assessment Regulations*. Projects required to be registered for EA are divided into two categories: Class I and Class II undertakings. A facility that extracts or processes metallic or non-metallic minerals, coal, peat, peat moss, gypsum, limestone, bituminous shale or oil shale, is identified as a Class I undertaking thus requiring registration for EA. The provincial EA process begins with the proponent presenting an overview of the mining project at a "One Window" Committee meeting attended by Nova Scotia Environment (NSE), Nova Scotia Labour and Advanced Education, NSDNR and any other regulators the government determines to be relevant based on the project specifics (potentially including federal regulators such as DFO, Environment and Climate Change Canada, and Transport Canada). The meeting is designed to inform the regulators of the project and for the regulators to advise the proponent on any possible regulatory issues from their respective departments. At the end of the meeting, the Department of Environment will inform the proponent if the project must be registered under the *Environmental Assessment Regulations*.

Following the "One Window" Committee meeting but prior to registering the EA, the proponent will usually meet with members of the One Window Committee and hold a project open house for the public so that potential topics of concern are addressed in the registration document. Once the proponent is satisfied that topics have been addressed, a finalized registration document is submitted to NSE for Registration. Within seven days of the project's registration, the proponent must publish a Notice of Registration to inform the public of the project, where copies of the EA Registration document can be reviewed, and to invite the submission of comments to the NSE EA Branch during the 30-day public review period. Copies of the document are also distributed by the Environmental Assessment Administrator to applicable regulators for review and comment. Following the public review and comment period, the Environmental Assessment Administrator summarizes comments received from the public and regulators, and submits to the Minister for an approval decision. From submission of the registration document to the Minister's decision is approximately 57 days (i.e., seven days for the document to be registered following submission and 50 days for the public and regulatory review period and the Minister's decision). However, the assessment may be extended if the Minister decides that more information, a focus report, or environmental assessment report is required. If it is an "Approval" decision, there are usually terms and conditions which must be addressed at Stage 2.

Federal

The requirements for federal EA are defined by the *Canadian Environmental Assessment Act*(CEAA) *2012*. CEAA 2012 applies to "Designated Projects," which are physical activities listed under the *Regulations Designating Physical Activities* under CEAA 2012. A project not listed under the *Regulations Designating Physical Activities* can still be designated by the Minister, if, in the Minister's opinion, the activity may result in adverse effects or public concern that may warrant the designation. Such a designation is unlikely for this project.

The *Regulations Designating Physical Activities* identify 48 "Physical Activities" that constitute Designated Projects which would require federal EA under CEAA 2012. Items 16 and 17 of the Schedule to the Regulations Designating Physical Activities include:

> ***16*** *The construction, operation, decommissioning and abandonment of a new*
> *(a) metal mine, other than a rare earth element mine or gold mine, with an ore production capacity of 3 000 t/day or more;*
> *(b) metal mill with an ore input capacity of 4 000 t/day or more;*
>
> ***17*** *The expansion of an existing*
> *(a) metal mine, other than a rare earth element mine or gold mine, that would result in an increase in the area of mine operations of 50% or more and a total ore production capacity of 3 000 t/day or more;*
> *(b) metal mill that would result in an increase in the area of mine operations of 50% or more and a total ore input capacity of 4 000 t/day or more;*

Since the mine will not exceed these thresholds, the Project is not considered a Designated Project under CEAA 2012, and a federal EA is not anticipated to be required. Further, as no aspect of the Project will be built on federal land, it is not expected that the mine will require an environmental assessment under Section 67 of CEAA 2012.

20.2.3 Stage 2 – Permits, Leases, Approvals

Stage 2 follows an EA Approval decision and involves the steps to attain the Permits, Leases, and Approvals required for mining activities. The three generally required are a Mineral Lease, Land Access Agreements, and an Industrial Approval. A Mineral Lease grants exclusive rights (20-year term) to some or all of the mineral resources in a specified area but does not allow any field activity beyond exploration. The approval time for a new lease is generally 60 days or less if all required information has been submitted. Land Access Agreements are a legally binding agreement that provides the Proponent access to the project area. The length of time required to acquire these agreements is variable.

An Industrial Approval (10-year term) is to construct, operate, or reclaim an open pit, milling facility, or bulk solids handling load out facility. The submission document is fairly substantial and if an Environmental Assessment was required, must address all of the terms and conditions outlined in the approval. Once the document has been submitted and determined to be "adequate" by the NSE, the approval process can take up to 60 days unless deficiencies are identified and the approval time period extended by the regulators.

20.2.4 Status of Permits and Licenses for the Project

A status summary of mine and facility permitting is outlined in Table 20-1, and in the Figure 20-1: Mine Permitting.

Table 20-1: Status of Permits and Licenses

Location	Permitting Status	Notes
Main Pit, Waste Rock Dump, Mill, Tailings Facility	• Mineral Lease 10-1, Industrial Approval, Environmental Assessment are approved. • Land Access Agreements in place.	• To address Land Access Agreements, ScoZinc has purchased all relevant land titles. • Indicated in purple hatching on Figure 20-1
Sheet Harbour	• Industrial Approval (pending renewal).	• Only an Industrial Approval is required for bulk solids handling load out facility. • Not indicated on Figure 20-1 based on location.
Southwest Expansion, Waste Rock Dump Expansions	• Mineral Lease 10-1, 12-1, 12-2, Environmental Assessment, and Industrial Approval are approved. • Land Access Agreements in place.	• To address Land Access Agreements, ScoZinc has purchased all relevant land titles. • Indicated in yellow hatching on Figure 20-1.

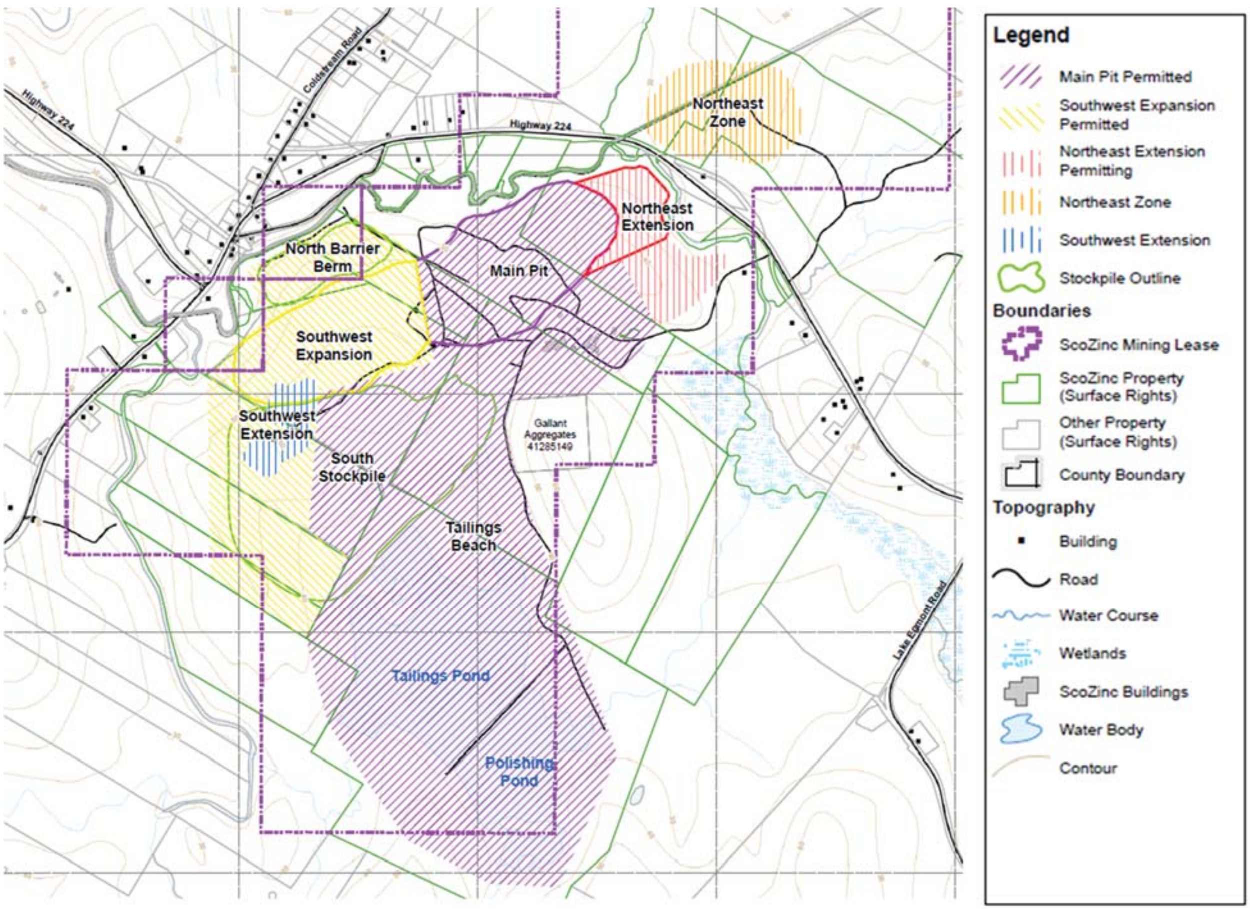
Location	Permitting Status	Notes
Northeast Extension	• Mineral Lease 10-1 approved. • Permitting has currently not been initiated. • Land Access Agreements in place.	• Will likely require an Environmental Assessment through the Provincial EA process; a Federal EA review is unlikely. The time estimate for a Provincial Environmental Assessment and Industrial Approval is expected to be 15 months for all permits to be in place. This can be done in conjunction with other expected Project expansions. • To address Land Access Agreements, ScoZinc has purchased all relevant land titles. • Indicated in red hatching on Figure 20-1.
Southwest Extension Pit	• Mineral Lease 10-1 approved. • Permitting has currently not been initiated. • Land Access Agreements in place.	• Will likely require an Environmental Assessment through the Provincial EA process; a Federal EA review is unlikely. The time estimate for a Provincial Environmental Assessment and Industrial Approval is expected to be 15 months for all permits to be in place. This can be done in conjunction with other expected Project expansions. • To address Land Access Agreements, ScoZinc has purchased all relevant land titles. • Indicated in blue hatching on Figure 20-1
Northeast Zone and Underground	• Mineral Lease 10-1 approved. • Permitting has currently not been initiated.	• Will likely require an Environmental Assessment through the Provincial EA process; a Federal EA review is unlikely. The time estimate for a Provincial Environmental Assessment and Industrial Approval is expected to be 15 months for all permits to be in place. This can be done in conjunction with other expected Project expansions. • To address Land Access Agreements, ScoZinc will likely need to purchase one relevant land title. • *Underground Mining Regulations* have filing requirements 90 days before proceeding with the development • Indicated in orange hatching on Figure 20-1

20.2.5 Shipping

Future deposit developments will use existing public roads that require no upgrading or infrastructure changes such as bridges. The primary route for the transport of zinc concentrates from the mill facility will be Highway 224 to Upper Musquodoboit and

Highway 277 to Sheet Harbour. All previous operations at the mine site used the same route for shipping zinc concentrate. The expected average daily number of trucks on this route (B-train styled with closed boxes) is four which is a small percentage (less than 2%) of the daily truck traffic based on recent data from public sources. Lead shipments are currently planned to be shipped through the Port of Halifax via Highway 224 to Highway 102 utilizing containers as the primary mode of transport. Only two to three containers per week are planned to be shipped to the Port of Halifax.

Figure 20-1: Mine Permitting



20.3 Waste, Water, and Site Monitoring

20.3.1 Waste Management

Tailings generated by the milling process will be pumped to the existing tailings storage facility (TSF). Dam raises will be required to establish the needed catchment size to accommodate the additional tailings volume generated by the project expansions. The costs to increase the capacity of the existing TSF are captured in the capital and operating budget for the operation. The tailings disposal plan will safely maximize usage of the existing storage area before raises occur.

Solid waste generated at the Project site will consist of unusable rock, organics and other naturally occurring materials stripped from the areas. Waste rock will be used, as appropriate, for infrastructure development with the excess being stored in waste rock stockpiles near the pit or backfilled into the mined-out portion of the pit. Lower-grade resource from the pits will be segregated and stockpiled for future processing at a later date. Gypsum excavated during pit stripping may be segregated and stockpiled for shipping off site. Gypsum is considered a commodity and there is a large gypsum mining operation about 15 km by road from the Project site.

Garbage produced on the mine site will be brought back to the existing facilities and trucked away for appropriate reuse or disposal to a provincially approved waste disposal facility.

All the administration, processing and support facilities will remain at the existing site location and are serviced by an existing on-site sewage treatment system.

20.3.2 Water Management

Waste water at the Project occurs in four management streams; tailings supernatant, pit water from pit dewatering, contact water from the mill and ancillary buildings, and contact water from waste rock stockpiles and other disturbed mining areas.

Tailings supernatant will be released to the existing tailings pond which is located as a component of the TSF at the toe of the tailings beach. Water for the milling process is recycled to the mill from this pond via pump and pipelines. Excess water in the tailings pond is discharged to the environment via an outflow structure where it flows to Annand Brook and from there to the Gays River upstream of the mine. Discharge through this structure is monitored for quality and volume by third party lab facilities and flow measuring devices. The water management process for the Project is consistent with the current operating parameters and permits for the Project.

Pit water from open pit dewatering is pumped and gravity fed to the tailings pond at the TSF. Similarly to tailings supernatant, these waters will eventually be recycled to the mill or discharge to the environment via Annand Brook.

Contact water from the mill and ancillary facilities is controlled by ditching and naturally flows towards the Main Pit or to a water storage pond outside the mine office building that in turn flows back to the pit. Water directed to the pit will be managed with the pit water and directed to the TSF. The water storage pond outside the mine office is intended as back-up fire suppression water for the operation.

Contact water from the North Barrier Berm will be captured in settling ponds located at the base of the berm so that the overall drainage and flow patterns leading into the existing catchment areas are maintained. Water will be discharged directly to the environment via pond spillways following tests to confirm it is within water quality limits. Contact water from the South Stockpile is directed to the TSF. In addition to the ponds, straw bales, straw waddles, and other sediment controls are used.

Water discharged from the tailings pond to the environment is currently subject to the Federal Metal Mining Effluent Regulations (MMER) approach and guidelines as well as some additional requirements from the Province via the existing Industrial Approval. Historically the operations discharges have met all requirements except minor copper exceedance that were properly reported, and corrective actions were successful.

20.3.3 Site Monitoring

Monitoring of the project site will be carried out by internal project staff with the assistance of qualified consultants. The cost for support staff, lab analysis and supporting external resources has been incorporated into the General and Administration (G&A) costs of the current Project economic model.

Site environmental monitoring anticipated to be required for the operation include:

- Hydrogeology – Groundwater monitoring wells have been and will be established at appropriate locations in the vicinity of Project facilities. Groundwater will be monitored for level and quality through the operations and closure phases of the Project. Precise locations and analysis parameters are set out in the current Industrial Approval.
- Surface Water – Surface water quality and flow volume monitoring has been and will be carried out at environmental discharge locations, in the receiving environment upstream and downstream of the discharge location where appropriate, and in other locations as required. The precise locations and analysis parameters of the monitoring program are established in the Industrial Approval.

- Wetlands – Areas planned for disturbance will be surveyed for the presence, size and quality of wetlands. A Wetlands Compensation Plan has been developed and will be carried out in accordance with Nova Scotia policy where required.
- Domestic Wells – Water wells on private property that fall within the perimeter prescribed for water well bonding will be tested for recharge and quality in advance of initiating operations (where allowed by the land owner). The CLC will provide a communications link between the Project General Manager and the local residents if domestic well impacts are alleged. Follow-up monitoring can be carried out and mitigation measures initiated if impacts are found.
- Blast Surveys – Prior to commencing operations, an updated survey of residences and other buildings within a prescribed perimeter of blasting operations will be carried out (where allowed by the land owner). Blasts will be monitored from several monitoring stations so that concussion and ground vibrations limits are not exceeded. The CLC will provide a communication link between the Project General Manager and local residents when blasting damage is alleged. Follow up inspections can be carried out in the event of a complaint and mitigation measures developed based on the outcome of these inspections.
- Dust - Total Suspended Particulate (TSP) emissions monitoring will be required once operations commence so that limits set out in the Industrial Approval are not exceeded. Particulate monitoring throughout different phases of the expansion process can be conducted using a Beta Array Monitor or High Volume Sampling. Air quality monitoring locations are set out in the Industrial Authorization.
- Noise – Ambient noise monitoring will be required during operations, so that sound levels to not exceed limits specified by the Industrial Authorization. Managing noise issues will be dependent on complaints received from local residents via the CLC in communication with the Project General Manager. Noise issues are likely to be episodic and associated with specific activities, locations, residence proximity, and climate conditions. Management of noise complaints will be case specific and depend on those attributes of the perceived issue.
- Archaeology - Personnel involved in all ground disturbances related to the construction and mining activities will be made aware of the potential for archaeological and/or cultural resources and the appropriate actions to take in identifying and reporting such features.
- Flora & Fauna - The monitoring program for the plant species *Hepatica nobilus* that was implemented by previous operators will be continued by ScoZinc, and a monitoring program for Wood Turtles is also required. Site specific habitat surveys required for expansion areas may identify additional species requiring monitoring and/or mitigation. The precise nature and type of these monitoring and mitigation programs will be prescribed in consultation with NSDNR.

- Socio-economic Parameters – The only socio-economic parameter likely to require future monitoring will be Traditional Land Use Surveys by First Nations. These surveys are typically done in advance of site disturbance to document plant and animal species in the area that are used by First Nations.

20.4 Reclamation

The needs and wishes of a community, as well as the mining process, may change as the project proceeds resulting in the requirement for a “Final Reclamation Plan” to be submitted within six months following the end of the extraction phase of the mine life. This plan is prepared by the proponent in consultation with the CLC, NSE, NSDNR and possibly other parties such as community groups or technical organizations. This “Final Reclamation Plan” is then approved and the proponent begins the work. The plan often includes monitoring components for aspects such as surface water quality, groundwater quality, water levels, vegetation growth and wetlands health. When the proponent completes all of the requirements of the Environmental Assessment, Industrial Authorization and any other reclamation-related conditions, the proponent is able to get back the reclamation bond value in full. Nova Scotia also allows for portions of the bond to be released if progressive reclamation is part of the project. For example, if 20 percent of the area has been reclaimed to the goal in the “preliminary reclamation plan”, a portion of that bond may be released if NSE and NSDNR are satisfied with the work completed.

In accordance with the above noted process, no final reclamation plan for the Project has been prepared or submitted. The requirements noted here are inferred from the currently accepted Preliminary Reclamation Plan which covers the Main Pit, tailings storage facility, waste stockpiles, and mine buildings. The current Preliminary Reclamation Plan was approved in July 2017.

Reclamation for the entire project will ultimately include the following:

- Removal of infrastructure and buildings
- Final rehabilitation of stockpiles
- Final surface contouring and sediment erosion control
- Assessment and remediation (if required) of any contaminated soils
- Rehabilitation of the former mining pits and tailings management area (including slope stabilization)
- Pit flooding
- Water level control
- Revegetation
- Monitoring

20.4.1 Post-Reclamation Monitoring

This section outlines monitoring specific to reclamation activities. The current Environmental Assessment and Industrial Authorization for Site Operations prescribe required monitoring for the duration of site operations that includes a number of aspects as described above (surface water, groundwater, rare plants, etc.). ScoZinc anticipates that in keeping with the currently approved reclamation plan, post-reclamation monitoring for the expanded project—including groundwater levels, surface water quality, vegetation, and aquatic habitat—will be carried out for a period of at least three years after final site reclamation.

Key elements of the reclamation plan may include the following:

- Vegetative Cover
  - Periodic inspections of the effectiveness of re-vegetation efforts will be needed. Areas identified as requiring additional effort will be noted, and a program to address the deficiencies in the re-vegetation will be developed and submitted to NSDNR and NSE for review.
- Slope and Shoreline Inspections
  - Slopes on stockpiles and shorelines of the lakes created by reclamation activities will be inspected for issues of erosion on a routine basis during reclamation operations. Inspections on a quarterly basis are proposed for the various Pits for a period of three years after pit closure or less as agreed to by NSDNR, NSE and the CLC. ScoZinc recognizes that additional monitoring may be required after the reclamation program is complete, if so directed by NSDNR and/or NSE.
- Pit Water Quality
  - Before decommissioning, the water being pumped from the Pits to the Tailing Management Area will be monitored for general chemistry and metals according to stipulations set forth in the Industrial Approval. Upon cessation of dewatering operations in the Pits, this monitoring will be replaced by seasonal water quality measurements from two depths (0-1 m and 1 m from bottom) in a central location of the pit lake for general chemistry and metals. An in-situ water quality meter may be used to provide a suite of parameters such as temperature, conductivity, and pH. It is proposed that monitoring continue for three years after the water level in the pit has reached stabilization and then be re-assessed by ScoZinc and NSE to determine if refinements to the program are required or cessation of the program is approved.
- Groundwater Levels and Quality

- The site is well equipped with monitoring wells that are used to address the current Industrial Approval requirements for both water level and water quality monitoring. It can be expected that additional monitoring wells will be required to address the future phases of the Project beyond the currently permitted project footprint. Available wells in this network will be monitored on a monthly basis for water level and for general chemistry and metals after mine closure. Each year, ScoZinc will review the data and consult with NSE on any required refinements to the program.

20.4.2 Reclamation Bonding

ScoZinc currently has a performance bond for the protection of domestic water supplies ($147,500) and a reclamation bond of $2.8 Million held with the Province of Nova Scotia. The domestic water supply related bond has been in place for over six years and has never needed to be drawn from due to an unresolved water supply related issue. The reclamation bond amount was calculated based on the Reclamation Plan submitted to and accepted by the Province in 2011. Bonding for the Southwest Expansion, permitted in 2012, is currently under negotiation with the Provincial Government. The initial cost estimate by the Province for the reclamation of this additional area was $3.7 Million. Two aspects of the bond; progressive bonding based on annual footprint expectations and the total amount of the bond, are currently under discussion. No additional bonding has been factored into the current economic model of the Project. It is expected that progressive reclamation of the historic project components (mined out pit areas and existing rock dumps and stockpiles) in conjunction with progressive bonding by the Government will allow the total bond requirements to be maintained at the current estimate of $6.3 Million as the Project progresses through the Northeast Extension, Northeast Pit, and Southwest Extension Pit phases.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

21.1.1 Capital Cost Summary

The projected initial capital cost is shown in Table 21-1. This is associated with the open pit mine and mill refurbishment only as capital for the underground operation begins in Year 5 as sustaining capital. Allowance for working capital has been made in the cashflow model. Included in the costs is also the capital for management team and core resources for startup (which has been included under the initial mine pre-stripping costs).

Table 21-1: Initial Capital Cost Summary (excluding Working Capital)

Item	Projected Capital Cost (thousands)
Environmental	$330
Site utilities and services	$360
Power system upgrades	$437
Administration building upgrades	$110
Mining (dewatering, technical service and minor equipment)	$1,396
Mine pre-stripping (with drilling and blasting, including capital for initial labor)	$12,746
Mill refurbishment	$8,296
Project direct and indirects	$1,918
Subtotal	$25,592
Contingency on non-quoted/calculated items (10%)	$1,285
Total	$26,877

The initial capital costs mainly consist of stripping and dewatering of the Main and Northeast open pit deposits, environmental expenses, mill refurbishment, and utilities and infrastructure upgrade to the site.

Power lines are to be realigned for the new open pit layout, which will require refurbishment of the existing power systems including relocation of the power line and installation of a new fibre optic feed to the site. Some general site maintenance will include bridge rehab, reclaim water system upgrade, and maintenance of the existing buildings. The initial capital towards environmental expenses encompasses the first year's surety bond[4], funds allocated for embankments construction near the

[4] An additional $200k is allocated for Year 2 and $250k for each year thereafter.

pits, recontouring of the site and addition of ditches. Extensive capital was allocated for mill refurbishment which is discussed in the following section.

Waste stripping will be performed by a contractor, including dewatering of the pit. ScoZinc undertook a very thorough pricing analysis for contract mining. All of the mining equipment other than an ancillary excavator, to be used around site as necessary, will be contractor owned including for underground mining.

21.1.2 Mill Capital Costs

The principal capital cost programs in the mill relate to the crushing, grinding, and concentrate filtering and plant automation processes.

The principal capital cost programs in the mill relate to the crushing, grinding and concentrate filtering and plant automation processes.

The crushing circuit has been one of the major impediments to stable and efficient operations, particularly during cold weather. The crushing circuit presented the greatest challenges to ScoZinc operations under the former management. A primary jaw crusher and radial stacker are proposed to help with feed, by improving the size of feed into the crushing circuit and grade control through blending. This with other changes and modifications will bring the current system to a three-staged crushing circuit, complete with a larger vibrating screen.

The first stage of the grinding circuit is the fine ore bin which feeds the grinding circuit by two parallel slot feeders from its base. The feeders had poor performance, were beyond repair and have been removed. ScoZinc proposes to mitigate historical problems of feeding from the fine ore bin through the use of six new variable drive vibrating feeders. Grinding improvements will increase the supply of process water to the mill, with automation to improve control, while flotation improvements are also gained through use of automation allowing better control of the reagents, including pH, on-stream analyzer for the critical process streams and flotation cell level control.

New feed wells are being installed for the Pb and Zn thickeners to improve control, with the Zn thickener also including new automated rakes to improve performance and capacity, while the former concentrate vacuum filters have been removed and scrapped. The rotary dryers for drying concentrate have been removed and placed into storage. The drying process will now be accomplished with two (2) vertical-plate filter presses. These units utilize a compressed air, high pressure water and filter cloth to remove the majority of the water from the respective Zn and Pb concentrates. The drying process will no longer require fuel oil thus reducing the overall mill emissions and reducing the costly use of fuel oil.

The capital plan was established to address known deficiencies or bottlenecks based on previous operations in 2008. The plant has been continuously maintained and some capital expenditures to support a restart have taken place starting in mid-2011. These funds are not included in the financial model.

21.2 Operating Costs

21.2.1 Open Pit Mine Operating Cost

The open pits will be developed and operated using conventional open pit mining practices and equipment with plans and designed layout in conformance with regulatory requirements.

Drilling, blasting, loading and haulage operation will be performed by a qualified and licensed contractor, under the direction of ScoZinc management. Refueling will not be covered by the contract and will be a cost incurred by ScoZinc.

The open pit mine operating cost components include:

- Mine operating labor costs;
- Subcontracted mining costs;
- Mine operating costs; and
- Mine indirect operating costs.

The total open pit mining cost is summarized in Table 21-2. The estimated annual mine operating costs vary in the cashflow model as material quantities and unit operating costs vary from year to year (with the monthly consideration given for the first two years, quarterly for the two years after and yearly for the remaining years). The mine operating costs include fuel, crushed stone, drilling and blasting service, dewatering and reclamation costs to support mine operation.

Table 21-2: Open Pit Mine Operating Cost Summary

Operating Cost (annual) (thousands):	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mine Labor	1,664	1,664	1,322	1,215	1,215	943	672	–	9,870
Contract Mining	18,798	27,310	41,781	22,891	21,417	30,191	8,412	–	170,799
General Mine Operations	7,354	13,560	13,902	13,345	7,387	15,961	1,967	390	73,866
Mine Indirects	350	350	350	350	350	350	350	350	2,800
Total Annual Operating Costs	28,146	42,864	57,355	37,801	30,369	47,716	11,672	1,412	257,335

ScoZinc staff continue to develop detailed operational plans based upon and guided by the plan developed for the updated PEA presented herein. These plans provide detailed haulage roads, ground control bench designs, design steps to address the underground works, and best disposal methods for overburden waste to minimize long and uphill loaded hauls.

21.2.2 Underground Mine Operating Cost

Planning for an underground mining operation targets a higher-grade resource located between the Main and Northeast pits, beneath the highway and Gays River. The underground workings and related facilities are designed to produce 500 t/d of higher grade feed to the mill to blend with the lower grade mill feed from the ongoing open pit operations. The total resource from underground is 289,302 t, which is a small percentage of the total feed to the mill – at just 4.4%.

The mechanized Cut-and-Fill mining method was selected for the project and has been carefully adapted and designed to suit the requirements of the ScoZinc underground project. This method was also selected in order to use uncemented backfill, thus minimizing operating costs and benefiting the project economics. The work will be conducted by contractors under management of ScoZinc personnel.

Underground mining operations are to commence in Year 5 of the overall life of mine schedule. About eight months of pre-production development will be capitalized as sustaining capital, largely in Year 5, followed by about 4 months of production providing high-grade feed to the mill. Year 6 will be a full production year and will see the completion of access development. About 3 months of production in Year 7 will complete the underground mining operation.

Underground capital and operating costs are presented in Table 21-3.

Table 21-3: Underground Capital[5] and Operating Cost Summary

Capital Cost (Sustaining)[4] (thousands):	Year 5	Year 6	Year 7	Total
Capital Development	3,649	286	-	3,935
Project Development	6,643	-	-	6,643
Mine Equipment	265	-	-	265
Salvage & Severance	-	-	-1,588	-1,588
Total Capital Cost	10,556	286	-1,588	9,254

Operating Cost (annual) (thousands):	Year 5	Year 6	Year 7	Total
Ore Extraction	1,447	3,947	817	6,211
Operating Development	2,006	1,566	200	3,771
General Mine Expense (incl. Major Repairs)	779	2,124	440	3,343
Total Annual Operating Costs	4,232	7,637	1,457	13,325

21.2.3 Mine Labor

The mine will be operated on two 12-hour shifts per day, 365 days per year basis with rotating crews. Labor supply within the province is more than ample to supply the needs of the operation. Labor rates used in this study are on par with current mining operations in the region.

21.2.4 Mill Operations and Processing Cost

A comparison of the actual 2008 Mill Process cost per dry metric tonne versus projected annual mill operating costs per dry metric tonne is shown in Table 21-4.

Table 21-4: Annual Mill Operating Cost Projections

Item	2008, January to September Cost (per dmt)	LOM Unit Cost (per dmt)	Difference
Total Processing Cost	$12.16	$13.46	$1.30
Power Cost	$2.39	$3.94	$1.55

In preparing the Mill Processing cost estimate, a detailed review of the 2008 actual costs were performed. Major changes in the mill, along with changes from the mine operations, were factored into the revised cost estimate. The impact of a 25% increase in mill through-put was factored into keys areas such as power. Other factors such as mill reagent costs were reviewed from 2008 and it was determined that purchases for the floatation process were impacted by excess inventory at the

[5] Includes 15% contingency

beginning of the year, thus requiring adjustments. A zero-base budget alternative was not prepared due to the significant advantage of utilizing the historic mill operating history to predict future cost when adjusted for proposed improvements to the process. However, the historical data was reviewed and analyzed to develop recoveries and adjusted reagent consumptions as indicated by 2008 metallurgical performance.

As can be seen from Table 21-4, the power cost is anticipated to be significantly higher due to increased rates and planned through-put increase. The overall energy cost and mill emissions will be reduced by the use of the pressure filter press for dewatering the concentrate product. Replacing the former disc filters and thermal dryers will eliminate the use of fuel oil to dry the product and the bottleneck caused by the dryer chute(s) plugging.

21.2.5 General and Administration Costs

The General and Administration (G&A) Costs are summarized in Table 21-5, where:

- Administration costs include projected management / administrative / support services labor costs as well as insurance, taxes, security, indirect equipment operating, office operating and consulting costs. Labor costs account for 68% of the administration cost in production Year 1. The mine indirect cost includes the mine staff labor cost during the pre-stripping phase.
- Safety and environmental costs include the coordinator labor cost, training and hygiene costs, environmental monitoring and contracted environmental services costs.
- Human resources costs include employee training, staff recognition, employee development and recruiting costs.

Table 21-5: General and Administration Costs

Area	Steady-State Annual Spending ($ thousands)	CAD per Tonne Mill Feed
Administration	2,299	4.04
Safety and Environmental	638	1.12
Human Resources	60	0.11
Total	2,997	5.27

22.0 ECONOMIC ANALYSIS

The potential economic viability of the Project was evaluated using a discounted cash flow analysis approach; monthly for the first two years, quarterly for the two years thereafter and annually for the remaining years of the Project. No inflation and escalation was applied in the cashflow model. In summary, using the Base Case metal pricing assumption (see Section 22.1) the results of the preliminary economic assessment indicate that:

- The Project has a mine life of approximately 7.7 years and offers an approximate 1.9 year payback.
- The Project has an estimated pre-tax internal rate of return (IRR) of 67.3% and an after-tax IRR of 63.7%.
- The Project has a pre-tax NPV of $159.9 million and an after-tax NPV of $127.9 million, both using a 5% discount rate. At an 8% discount rate, the pre-tax NPV is $134.2 million and the after-tax NPV is $107.7 million.
- The Project has an average C1 Zn cash cost of production of C$0.73 (US$0.59) per pound of Zn over the planned life of the operation (after deducting credits for Pb).

This preliminary economic assessment is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

22.1 Input Parameters

The input parameters to the cashflow model are listed in Table 22-1. All amounts are expressed in Canadian dollars, except where noted.

Table 22-1: Cashflow Model Parameters

Base Case Zn price:	$US 1.25/lb (for life of mine)
Base Case Pb price:	$US 1.05/lb (for life of mine)
US:CDN exchange rate:	$US 0.81 = $CDN 1.00
Zn mill recovery (life of mine)	86%
Zn Treatment Charge:	$70 per tonne of concentrate (Year 1)
	$100 per tonne of concentrate (Year 2)
	$150 per tonne of concentrate (Year 3)
	$190 per tonne of concentrate (Year 4-8)
Zn Concentrate Grade:	57%
Zn Concentrate Moisture	9%
Zn Payable from Smelter	85%
Land Freight:	$11.61/tonne
Ocean Freight:	$30/tonne
Pb mill recovery (life of mine)	85.7%
Pb Treatment Charge:	$70 per tonne of concentrate (Year 1)
	$90 per tonne of concentrate (Year 2)
	$135 per tonne of concentrate (Year 3)
	$150 per tonne of concentrate (Year 4-8)
Pb Concentrate Grade:	71%
Pb Concentrate Moisture	7%
Pb Payable from Smelter	95%
Land Freight:	$11.61/tonne
Liners:	$70 per container
Ocean Freight:	$30/tonne
Capital Cost Excluding Working Capital:	$26.9 million
Annual Inflation or Escalation Included	$0
Salvage Value for Mill and Equipment on Final Closure	$0

22.2 Results

The results of the economic analysis are as follows:

- The Project has a mine life of approximately 7.7 years with a project payback of approximately 1.9 years.
- Total payable metal production over the life of the project is projected to be 323 million pounds (146,700 t) of Zn and 184 million pounds (83,482 t) of Pb.
- Total LOM gross revenue is about $645 million, of which 67% is derived from Zn and 33% derived from Pb.
- The life of mine C1 Zn cash cost of production is C$0.73 (US$0.59) per pound of Zn (after deducting credits for Pb).

The economic results are summarized in Table 22-2 and the cash flow model showing the annual sub-totals is provided in Table 22-5.

Table 22-2: NPV and IRR Summary

	Pre-tax	After-tax
NPV (5%)	$159.9M	$127.9M
NPV (8%)	$134.2M	$107.7M
IRR	67.3%	63.7%

This preliminary economic assessment is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

22.3 Sensitivities

The economics of the project are most sensitive to exchange rate, metal prices, the grade of the potentially mineable mineralization, capital and operating costs. Two set of sensitivity analysis was conducted one showing the Zn price escalating while Pb prices holds steady, representative of expectations in 2018, while the second set of analysis shows prices escalating by US$ 0.10/lb with US$ 0.20/lb difference between the Zn and Pb prices. Table 22-3 and Table 22-4 presents the results of the metal price sensitivities analysis for the first and second cases respectively. The results of the sensitivity analysis are shown in Figure 22-1 (5% discount rate case) and Figure 22-2 (8% discount rate case), with the metal prices for both Case 1 and Case 2 included. Exchange rate, average grade, capital and operating costs are not affected by the change in Pb pricing between the two cases.

Table 22-3: Metal Price Sensitivities Analysis, Case 1: Zinc Prices Escalating with Lead Prices Holding at US $1.15/lb.

Zinc/Lead Price US$/lb	NPV Pre-Tax		NPV After-Tax		IRR %		Payback Period	Average Annual EBITDA
	NPV 5%	NPV 8%	NPV 5%	NPV 8%	Pre-Tax	After-Tax		
0.95/0.75	7.2M	-1.2M	7.2M	-1.2M	7.5%	7.5%	6.54	8.1M
1.05/0.85	58.1M	43.9M	51.1M	38.2M	25.8%	24.3%	4.98	15.8M
1.15/0.95	109.0M	89.1M	89.0M	72.7M	45.5%	42.7%	3.25	23.4M
1.25/1.05*	**159.9M**	**134.2M**	**127.9M**	**107.7M**	**67.3%**	**63.7%**	**1.91**	**31.1M**
1.35/1.15	210.8M	179.4M	169.1M	144.6M	91.3%	87.9%	1.56	38.8M
1.45/1.15	243.4M	208.4M	197.0M	169.6M	109.2%	106.3%	1.41	43.7M
1.55/1.15	276.1M	237.5M	224.9M	194.7M	128.5%	126.1%	1.22	48.6M
1.65/1.15	308.8M	266.5M	251.3M	218.3M	149.1%	146.3%	1.10	53.5M
1.75/1.15	341.4M	295.6M	270.7M	235.6M	171.1%	164.3%	1.00	58.4M

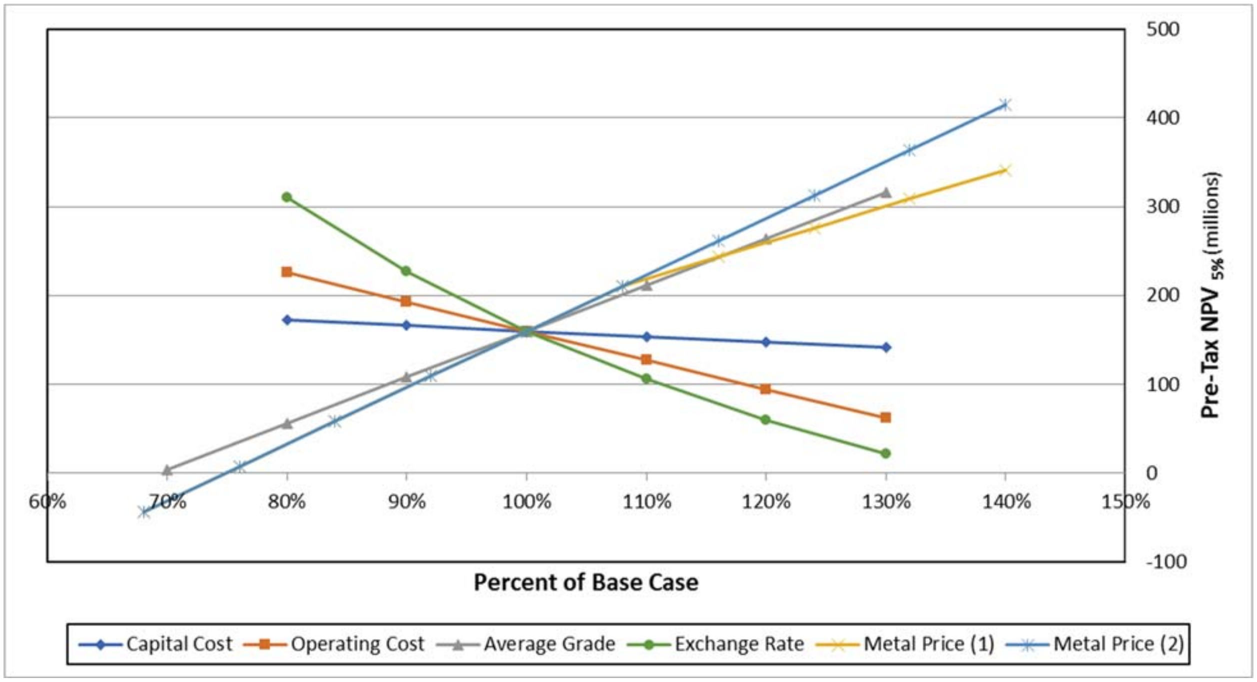
* Base Case

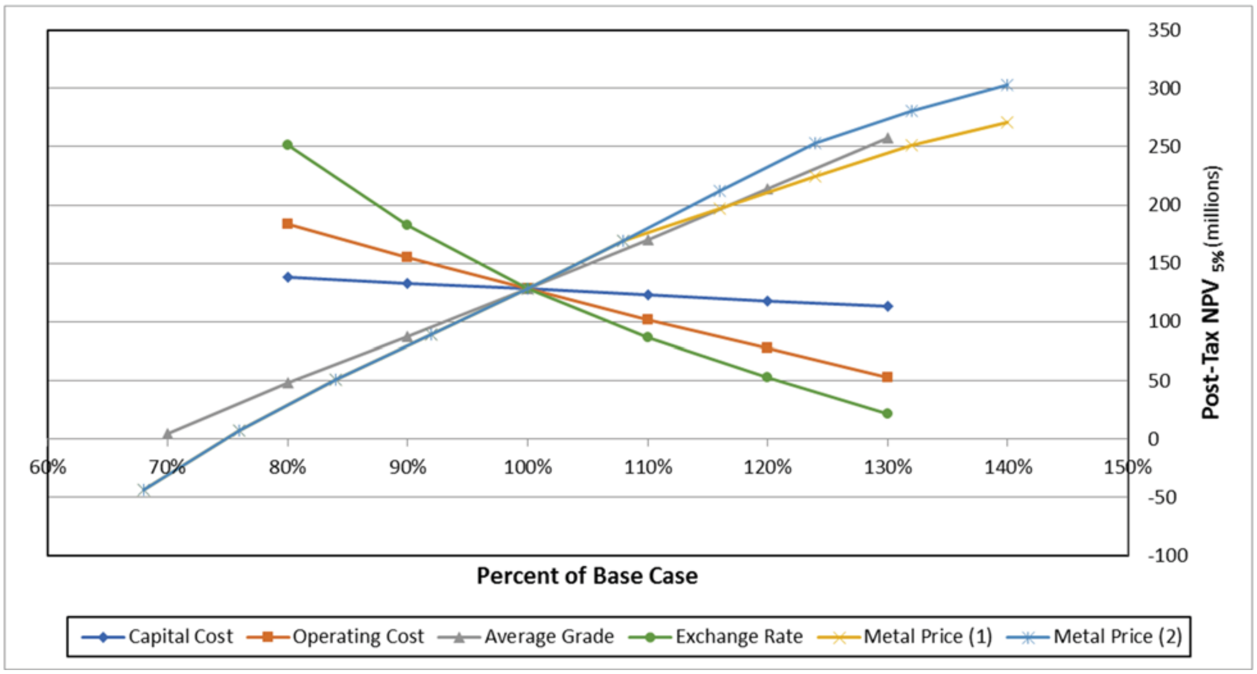
Table 22-4: Metal Price Sensitivities Analysis, Case 2: Both Zinc and Lead Prices Escalating

Zinc/Lead Price US$/lb	NPV Pre-Tax		NPV After-Tax		IRR %		Payback Period NPV 8%	Average Annual EBITDA
	NPV 5%	NPV 8%	NPV 5%	NPV 8%	Pre-Tax	NPV 5%		
0.95/0.75	7.2M	-1.2M	7.2M	-1.2M	7.5%	7.5%	6.54	8.1M
1.05/0.85	58.1M	43.9M	51.1M	38.2M	25.8%	24.3%	4.98	15.8M
1.15/0.95	109.0M	89.1M	89.0M	72.7M	45.5%	42.7%	3.25	23.4M
1.25/1.05*	**159.9M**	**134.2M**	**127.9M**	**107.7M**	**67.3%**	**63.7%**	**1.91**	**31.1M**
1.35/1.15	210.8M	179.4M	169.1M	144.6M	91.3%	87.9%	1.56	38.8M
1.45/1.25	261.7M	224.5M	212.3M	183.3M	117.6%	114.9%	1.33	46.5M
1.55/1.35	312.6M	269.7M	252.8M	219.5M	146.1%	142.7%	1.14	54.1M
1.65/1.45	363.5M	314.8M	280.4M	243.9M	176.7%	166.3%	1.00	61.8M
1.75/1.55	414.4M	360.0M	303.4M	264.0M	209.4%	190.6%	0.90	69.5M

* Base Case

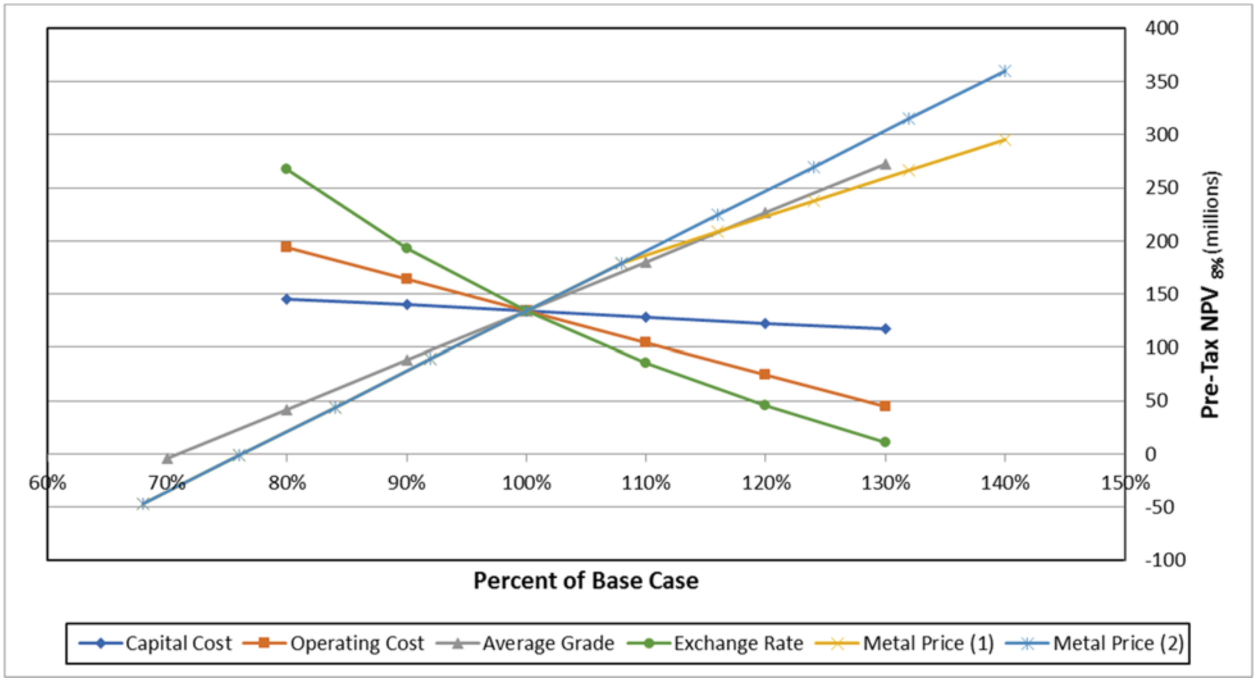
Figure 22-1: NPV5% Sensitivity





The results outlined in the sensitivity analysis shows that the project is most sensitive to metal pricing, average grade and the exchange rate, since these have the steepest trends, and is mostly due to the increase or decrease in those parameters have direct effect on the gross revenue. Capital cost appears to be the least sensitive parameter, with a 50% change in capital producing minor change in the net revenue.

Figure 22-2: NPV$_{8\%}$ Sensitivity



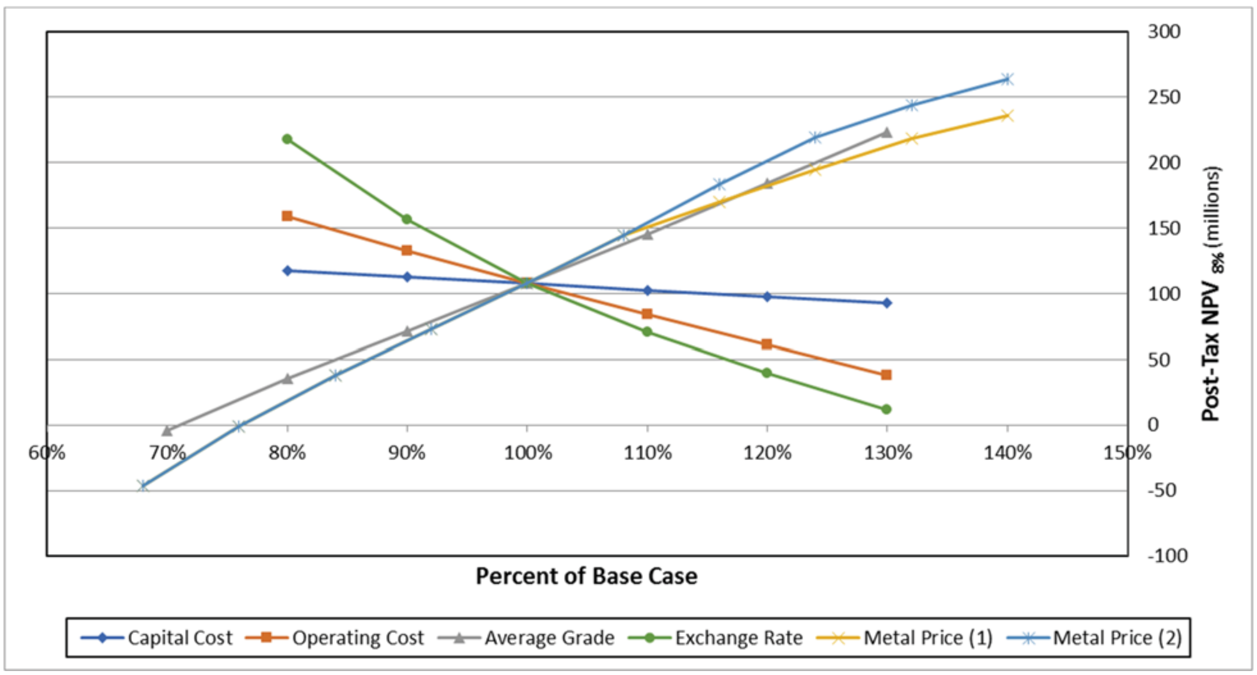


Table 22-5: Cashflow Model Detail

Profit & Loss	(Cdn $'s)	Year 1 Sub-Total	Year 2 Sub-Total	Year 3 Sub-Total	Year 4 Sub-Total	Year 5 Sub-Total	Year 6 Sub-Total	Year 7 Sub-Total	Year 8 Sub-Total	LOM Total
Ore to Mill *(tonnes)*		**568,341**	**851,939**	**852,816**	**852,816**	**852,816**	**852,816**	**867,319**	**852,816**	**6,551,679**
Tonnes per day		1,730	2,593	2,596	2,596	2,596	2,596	2,640	2,596	2,216
Zinc Head Grade	%	2.51	3.00	2.98	3.45	3.17	3.55	4.04	1.62	3.06
Lead Head Grade	%	0.67	1.28	1.80	1.66	1.48	2.18	2.07	1.07	1.57
From Open Pits (tonnes):		450,341	350,708	348,978	572,030	357,275	410,146	829,245	150,800	3,469,524
Tonnes per day		1,371	1,068	1,062	1,741	1,088	1,249	2,524	459	1,320
Zinc Head Grade	%	2.49	3.48	2.60	4.08	2.45	1.58	3.90	1.62	3.05
Lead Head Grade	%	0.76	1.71	1.79	2.09	1.03	1.37	1.98	1.07	1.58
From Stockpiles (tonnes):		118,000	501,230	503,838	280,787	428,138	258,845	-	702,016	2,792,853
Tonnes per day		359	1,526	1,534	855	1,303	788	-	2,137	945
Zinc Head Grade	%	2.60	2.66	3.24	2.16	3.26	4.55	-	1.62	2.72
Lead Head Grade	%	0.34	0.99	1.81	0.77	1.53	2.37	-	1.07	1.32
From Underground (tonnes):		-	-	-	-	67,403	183,825	38,074	-	289,302
Tonnes per day		-	-	-	-	205	560	116	-	294
Zinc Head Grade	%	-	-	-	-	6.35	6.52	7.00	-	6.54
Lead Head Grade	%	-	-	-	-	3.63	3.73	4.01	-	3.74
Zinc Concentrate	tonnes in Con	21,556	38,549	38,288	44,329	40,751	45,626	52,843	20,845	302,787
Lead Concentrate	tonnes in Con	4,602	13,197	18,539	17,042	15,284	22,441	21,650	11,014	123,769
Zinc Recovery	%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%
Lead Recovery	%	85.7%	85.7%	85.7%	85.7%	85.7%	85.7%	85.7%	85.7%	85.7%
Recovered Zinc	t	12,287	21,973	21,824	25,268	23,228	26,007	30,121	11,881	172,589
Recovered Lead	t	3,267	9,370	13,163	12,100	10,851	15,933	15,372	7,820	87,876
Metal Payable from Smelter - Zinc	%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Metal Payable from Smelter - Lead	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Payable Zinc	lbs.	23,024,618	41,175,720	40,896,721	47,349,524	43,527,901	48,735,236	56,443,826	22,264,937	323,418,483
Payable Lead	lbs.	6,842,804	19,624,409	27,567,537	25,341,175	22,727,138	33,370,335	32,194,257	16,378,619	184,046,274
Zinc Price	$ / lb.	$1.54	$1.54	$1.54	$1.54	$1.54	$1.54	$1.54	$1.54	$1.54
Lead Price	$ / lb.	$1.30	$1.30	$1.30	$1.30	$1.30	$1.30	$1.30	$1.30	$1.30
Zinc Revenue		35,531,818	63,542,778	63,112,223	73,070,254	67,172,687	75,208,697	87,104,669	34,359,471	499,102,598
Lead Revenue		8,870,302	25,439,049	35,735,696	32,849,671	29,461,105	43,257,841	41,733,296	21,231,543	238,578,504
Revenues from Operations		**44,402,120**	**88,981,827**	**98,847,919**	**105,919,925**	**96,633,791**	**118,466,539**	**128,837,965**	**55,591,014**	**737,681,101**
TC/RC's & Freight - Zinc		2,643,788	5,998,825	8,061,906	11,282,465	10,371,847	11,612,653	13,449,459	5,305,299	68,726,244
TC/RC's & Freight - Lead		566,108	1,978,299	3,576,399	3,562,433	3,194,955	4,691,163	4,525,831	2,302,487	24,397,675
Gross Revenue		**41,192,224**	**81,004,703**	**87,209,614**	**91,075,027**	**83,066,989**	**102,162,723**	**110,862,675**	**47,983,228**	**644,557,182**
Provincial Royalty		823,844	1,620,094	1,744,192	1,821,501	1,661,340	2,043,254	2,217,253	959,665	12,891,144
Net Revenue		**40,368,379**	**79,384,609**	**85,465,422**	**89,253,526**	**81,405,650**	**100,119,468**	**108,645,421**	**47,023,564**	**631,666,039**
Operating Expenses - Open Pit Mine		28,145,892	42,863,887	57,355,355	37,800,543	30,369,320	47,716,068	11,672,166	1,412,196	257,335,426
Operating Expenses - Underground Mine		0	0	0	0	4,231,543	7,636,380	1,456,988	0	13,324,911
Operating Expenses - Mill		7,957,848	11,722,744	11,722,876	11,722,876	11,722,876	11,722,876	11,722,876	9,892,030	88,187,004
Total Operating Expenses		36,103,740	54,586,631	69,078,231	49,523,420	46,323,740	67,075,324	24,852,030	11,304,225	358,847,341
Gross Profit		**4,264,639**	**24,797,978**	**16,387,191**	**39,730,106**	**35,081,910**	**33,044,144**	**83,793,391**	**35,719,338**	**272,818,698**
ScoZinc SG&A		2,996,853	2,996,853	2,996,853	2,996,853	2,996,853	2,996,853	2,996,853	2,869,353	23,847,324
EBITDA		**1,267,786**	**21,801,125**	**13,390,338**	**36,733,253**	**32,085,057**	**30,047,291**	**80,796,538**	**32,849,985**	**248,971,374**
Net Change in Working Capital		(615,971)	(2,593,081)	3,530,260	(5,338,008)	11,583,177	136,097	(4,185,472)	4,065,135	6,582,137
Property Capital		0	(264,000)	0	0	0	0	0	0	(264,000)
Underground Capital		0	0	0	0	(11,611,991)	(314,050)	1,746,563	0	(10,179,478)
Restart/sustaining capital		(26,876,998)	(220,000)	(275,000)	(275,000)	(275,000)	(275,000)	(275,000)	(275,000)	(28,746,998)
Cash flow before taxes		(26,225,183)	18,724,045	16,645,597	31,120,246	31,781,243	29,594,339	78,082,629	36,640,120	216,363,035
Income taxes (payable) refund		0	0	0	0	(5,687,435)	(7,663,803)	(23,847,721)	(7,006,166)	(44,205,125)
Cash flow for Debt Servicing		**(26,225,183)**	**18,724,045**	**16,645,597**	**31,120,246**	**26,093,807**	**21,930,536**	**54,234,908**	**29,633,954**	**172,157,910**

		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	LOM
Mine Operating Cost/Tonne Milled		$ 49.52	$ 50.31	$ 67.25	$ 44.32	$ 40.57	$ 64.91	$ 15.14	$ 1.66	$ 41.31
Mill Operating Cost		$ 14.00	$ 13.76	$ 13.75	$ 13.75	$ 13.75	$ 13.75	$ 13.52	$ 11.60	$ 13.46
Total Cost/Tonne Milled		$ 63.52	$ 64.07	$ 81.00	$ 58.07	$ 54.32	$ 78.65	$ 28.65	$ 13.26	$ 54.77
Strip Ratio		15.0	8.6	21.7	8.1	23.7	11.9	3.8	-	12.0
Total Cost per Ton Mined All Materials		$ 1.97	$ 3.25	$ 2.68	$ 3.66	$ 3.29	$ 3.72	$ 3.17	$ -	$ 3.05
C1 Cash Cost (Lb. Zn Metal) (CDN)		**$ 1.45**	**$ 0.97**	**$ 1.17**	**$ 0.73**	**$ 0.77**	**$ 0.88**	**$ 0.07**	**$ 0.02**	**$ 0.73**
C1 Cash Cost (Lb. Zn Metal) (USD)		**$ 1.18**	**$ 0.79**	**$ 0.95**	**$ 0.59**	**$ 0.62**	**$ 0.72**	**$ 0.06**	**$ 0.02**	**$ 0.59**
Total Cost (operating+capex+sustaining)	CDN	$ 2.62	$ 0.99	$ 1.18	$ 0.73	$ 1.04	$ 0.90	$ 0.05	$ 0.04	$ 0.85
Total Cost (operating+capex+sustaining)	USD	$ 2.34	$ 0.80	$ 0.96	$ 0.60	$ 0.90	$ 0.73	$ 0.03	$ 0.03	$ 0.72

23.0 ADJACENT PROPERTIES

There are no adjacent properties considered in the report.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information that we are aware of that has not been presented in the other sections of this report.

25.0 INTERPRETATION AND CONCLUSIONS

The Gays River Deposit, consisting of the Main and Northeast Zone deposits, define a shallow Zn-Pb-mineralized zone has been outlined over strike length of almost four kilometers. Outcrops are rare, but the deposit sub-crop under the unconsolidated glacial till overburden. The dolostone host rock drapes over a paleo-shoreline of metasediments at dips that vary between 30°-40° and vertical, averaging 40°-60°. Thickness varies from less than 1 m to over 10 m in true thickness.

The Zn is contained in a very low-iron sphalerite that is highly marketable.

Mineral resources were identified in Measured, Indicated, and Inferred categories. For the Gays River Deposit, in both the Main and Northeast Zones, Measured plus Indicated mineral resources totaled 5.77 million tonnes with average grades of 3.00% Zn and 1.56% Pb. Inferred mineral resources in the designed pits totaled 0.63 million tonnes with average grades of 2.53% Zn and 1.48% Pb. The block cutoff grade was 0.75% Zn-equivalent.

Due to the inclusion of less than 10% Inferred material in the production schedule, this report is presented as a PEA; however, this study is largely founded on historical production records, historic engineering, budgetary cost quotations, and in some instances, cost quotations providing additional confidence in the study and financial model assumptions. The Base Case economics of the two Gays River pits are robust with a post-tax NPV (discounted at 8%) of $107.7 million, IRR of 63.7%, and payback in 1.91 years, based on Base Case metal price assumptions of US$1.25 per pound for Zn and US$1.05 per pound for Pb.

For the Gays River Deposit, some of the identified mineral resources are located underneath Gays River between the proposed Main and Northeast pits, including considerable higher grade mineralization. Sandy soil lies underneath Gays River, so mining close to the river could be susceptible to water inundation. Additional mineral resources that lie close to, or underneath Gays River, would be relatively more expensive to recover due to the added cost of either:(a) diverting the river and mining by open pit, or (b) recovering the higher grade portions using underground mining methods. The latter is the most practical approach after the Main and/or Northeast pits are established and pit dewatering having drawn down the local water table.

The two conventional open pits and the proposed underground mine will provide a blended feed to the mill. Production scheduling is based on an average production rate of 852,800 t/a (or 2,600 t/d) into the mill over an average of 328 operating days per year. The average waste-to-resource ratio for the LOM open pits is 12-to-1 (excluding pre-stripping which is included in the capital costs). Approximately 62% of the waste is readily removed without blasting, including soils that will be used for reclamation, and 22% of the waste is gypsum, which will be stockpiled for possible future sale: no value for gypsum has been used in the PEA. Open pit mine dilution and mining losses are assumed to be 10% and 5%, respectively. The material movement rate, including resource and waste, in the 7.6-year production schedule peaks at approximately 53,000 tpd. In-pit diluted mineral resources are 6,552,000 t grading 3.03% Zn and 1.59% Pb.

An underground operation based on Cut-and-Fill mining with un-cemented backfill, producing up to 500 tpd, with an average of 294 tpd of higher grade mill feed from the higher-grade zone between the Main and Northeast pits is included in this PEA. A drawdown of the water table in the proposed mine area would be achieved by a combination of open pit and dewatering wells. The development of the underground mine access requires a sustaining capital investment of about $11.9million including contingency, most within Year 5 of the overall mining schedule to develop access to the higher-grade zones. Diluted and recoverable underground mineral resources are estimated at 289,000 kt grading 6.54% Zn and 3.74% Pb. This material will be blended with open pit and stockpile feed to the mill over approximately 2 years beginning in the second half of Year 5 of the LOM plan.

Aggregate production from the two open pits and the underground mine is estimated at 6,552,000 t grading 3.06% Zn and 1.57% Pb.

Table 25-1 summarizes the results of the current PEA in comparison with the 2013 PEA.

Table 25-1: Comparison of Results Between Current and Previous PEA

	Previous PEA (Nov 22, 2012 news release)	Current PEA
Mill Processing Rate (tonnes per day)	2,500	2,600
Unit Operating Costs (per tonne milled) for LOM	$40.84	$54.77
Restart Capital (including contingency and working capital)	$32.8 M	$27.4 M
Zn Price	$1.00 (USD)	$1.25 (USD)
Pb Price	$1.10 (USD)	$1.05 (USD)
Exchange Rate (CAD to USD)	0.98	0.81
Pre-Tax NPV (at 5%)	$61.30	$159.90
Pre-Tax NPV (at 8%)	$52.40	$134.20
After-Tax NPV (at 5%)	$51.90	$127.90
After-Tax NPV (at 8%)	$44.40	$107.70
Pre-Tax Internal Rate of Return	49.0%	67.3%
After-Tax Internal Rate of Return	46.2%	63.7%
Payback	1.56 years	1.91 years
Zinc C1 Cash Cost for LOM	$0.51 (CAD) $0.50 (USD)	$0.73 (CAD) $0.59 (USD)
Annual Average EBITDA for first five years	$24.1 M	$32.5 M
Zinc Treatment Charge	$190	Yr 1 $70 Lt $190
Pb Treatment Charge	$100	Yr 1 $70 Lt $150

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Addressing metallurgical performance risks, a small underground operation, updated equipment costs, and revised assumptions demonstrate encouraging economics for the project (Table 25-1).

The Gays River deposits (Main and Northeast Zones) and the ScoZinc facilities are prepared for final refurbishment and restart of mining operations.

26.0 RECOMMENDATIONS

The objective of the following recommendations is to improve the certainty of achieving and expanding the economics forecasted in this study. Recommendations from the report dated 08 October 2012, titled "Updated Mineral Resource Report for the Gays River and Getty Deposits" (available on SEDAR) are included herein for continuity and clarity.

26.1 Geology

In advance of any further drilling being done on the Gays River deposits, the Northeast Zone should be revisited and remodeled under a similar cutoff grade to the work done during this study for the Main Zone and its Southwest expansion. Previous work used a 0.5% Zn-equivalent cut-off above 100 m and a 2.0% Zn-equivalent cutoff below. It is assumed that, given the positive results of the updated resource modeling on which this study is based, further mineralization could be identified through more detailed analysis, thereby better defining the mineralizing system that will allow for a more accurate assessment for future drilling. The remodeling work is estimated to cost at least $50,000.

ScoZinc should re-examine the RQD and RMR geomechanical data collected in the 2011 drilling program and use it to better define criteria for the physical properties of the host rocks to the Gays River deposit. This geomechanical assessment is estimated to cost $40,000.

26.2 Mining

Detailed geotechnical investigations should be performed to examine the geometry and stability of the open pit walls during dewatering and planned mining operations. The work should identify appropriate final wall designs to ensure safe operations. This work is estimated to cost between $60,000 and $80,000.

It is recommended that detailed mine planning and mine schedule optimization be carried out by qualified engineers, taking into consideration the historic underground workings, geotechnical stability, worker and visitor safety, regulatory requirements, existing underground survey records, pit bench layout, and stability monitoring aspects. As part of the detailed mine planning work, a screening level risk assessment should be used to assess the possible need for additional engineered controls to eliminate or mitigate associated potential risks in the open pits, along the haul routes, and at the mine material stockpile locations. Engineered controls are measures or procedures proactively designed to eliminate or mitigate risks. This work will be performed by in-house technical staff supported by external consultants and is estimated to cost between $80,000 and $100,000.

The metal prices of Zn and Pb used in this study are different than those used in the resource model equivalent grade calculation. It is recommended that the block model be updated based on new prices.

Complete the pre-stripping and pit rehab to establish feed for the mill re-start at an estimated cost of $13.1 million.

26.3 Metallurgy

The proposed infrastructure and mill improvements should be implemented prior to the re-start. This will serve as a basis for defining the scope of ongoing investigations for effective mill improvements. The highest priority changes include the following items.

- Complete a realignment of the power systems to eliminate interference with the open pit mine plan.
- Install primary crushing capability, as well as stockpiling of crushed materials to improve crushing plant capacity and mill feed grade control.
- Improve the crushing circuit through introduction of tertiary crushing and product sizing by including a double deck screen.
- Upgrade fine ore bin feeders and controls for improved plant reliability and rod mill feed performance.
- Introduce automation into the fine ore, grinding, and flotation circuits.
- Add an on-stream analyzer and reagent controls to improve flotation performance.
- Upgrade the Zn thickener.
- Replace the vacuum filters and dryers with plate filters.
- Increase the reclaim water / process water system capacity for higher throughput.
- Upgrade the open pit dewatering pumping and well system to address historic water ingress issues.
- Update the closure bond to reflect current plans.
- Miscellaneous capital for buildings, infrastructure, and re-start.

The budget for all the expenditures is estimated to be $17-18 million.

It is recommended that ScoZinc plan to conduct plant process surveys as soon as reasonable circuit stability has been achieved, in addition to the following recommendations:

- Implement effective crew training programs prior to plant commissioning.
- Ensure the assay laboratory, metallurgical laboratory, and on-stream sampling / analysis systems are commissioned, to the maximum extent possible, prior to the resumption of operations.
- Arrange to send samples of intermediate and exit flotation circuit products to a qualified laboratory for mineralogical analyses, once reasonable circuit stability has been achieved.
- Conduct bench-scale flotation tests at the mine site laboratory on composite samples of mill feed. By so doing, the effects of grind, regrind, retention times, and other key variables can be rapidly determined. Plant results can be compared with the best results achieved in the laboratory to provide an indication of potential improvements in plant performance.
- Investigate the potential application of SAG milling to reconfigure the current crushing and grinding circuits.

This work is included in the restart capital cost and the first year of operating costs.

26.4 Gypsum

It is recommended that ScoZinc review existing information on the quantity and quality of gypsum rock and assess market opportunities for gypsum sales. The present preliminary economic assessment assumes that gypsum rock produced by the pit waste stripping operations would be disposed of and separated in the mine waste stockpile. This will largely be an internal assessment but may require external market assistance. The work is estimated to cost $20,000 to $30,000.

26.5 Getty Deposit

26.5.1 Property Description and Location

The Getty Deposit is located northwest of the Gays River Deposit on the western side of Gays River. The two deposits are separated by less than one kilometer. Access to the property is by paved roads and is approximately fifteen kilometers off of the Trans-Canada Highway along Route 224. The Halifax International Airport is located twenty kilometers southwest of the mine site.

The resources for the Getty deposit are included in this section; however, the deposit was excluded from the economic assessment in this study. It presents a significant mine expansion potential beginning in year 8 and will be evaluated in the future.

26.5.2 Land Tenure

Regarding the Getty Deposit, Cullen et al. (2011) stated that "in September, 2006 the provincial government tendered exploration rights to the closed Getty property and Exploration Licenses 6959 and 6960 were subsequently issued to Acadian on October 20th, 2006 as the successful bidder under the tendering process."

ScoZinc Limited currently holds the mineral rights to the Gays River and Getty Deposits, as well as the mining rights and surface rights for Scotia Mine (ScoZinc Operations/Gays River Deposit). The existing surface rights are sufficient for currently planned mining operations.

26.5.3 Getty Deposit Mineral Resource Estimate

Cullen et al. (2011) summarized their resource estimate of the Getty Deposit (see Table 26-1) as follows:

> *"The estimation of mineral resources of the Getty deposit is based on 138 drill holes completed by Acadian in 2007 and 2008 and 184 historic drill holes completed during the 1970's by prior operators. Getty Northeast Mines Limited drilled 181 of these historic drill holes and the remaining 3 drill holes were completed by Imperial Oil Limited. It should be noted that Mercator managed the 2007 and 2008 drilling programs for Acadian and that Quality Control and Quality Assurance protocols included the systematic insertion of independent analytical standards and blanks plus duplicate sample analyses and independent check sample analyses."*

Table 26-1: Getty Deposit Mineral Resources

Resource Category	Zn Eq. % Cut-off	Tonnes (Rounded)	Zinc %	Lead %	Zinc Eq %[1]
Measured	2.00	1,550,000	1.97	1.45	3.68
Indicated	2.00	2,810,000	1.82	1.44	3.51
Indicated + Measured	**2.00**	**4,360,000**	**1.87**	**1.44**	**3.57**
Inferred	**2.00**	**960,000**	**1.73**	**1.59**	**3.60**

Zinc Equivalent % (Zn Eq.%) = Zn % + (Pb % x 1.18) and is based on mill recoveries of 89.3% for zinc and 89.5% for lead, $US1.10/lb Zn and $US1.15/lb Pb metal pricing and smelter returns of 85% for Zn and 95% for Pb. Reliance On Other Experts

This report was prepared by ScoZinc and Selwyn; the material, conclusions and recommendations contained herein are based upon information available to ScoZinc and Selwyn at the time of report preparation.

ScoZinc and Selwyn consulted several experts during the writing of this report; Wood Mackenzie, MineTech International Limited, Conestoga-Rovers & Associates, Atlantic Caterpillar, Hewitt Caterpillar and Caterpillar Global Mining. ScoZinc and Selwyn have no reason to question the quality or validity of the data and opinions expressed by these experts. ScoZinc and Selwyn supports the data and conclusions of those qualified persons who have been included in this report.

This report includes opinions that concern exploration and development potential for the project as well as recommendations for further analysis. These are intended to serve as guidance and should not be taken as a guarantee of success.

26.6 Property Description and Location

The Getty property consists of 62 contiguous mineral claims, approximately 992 hectares. The location of the deposit is shown in Figure 26-1 and Figure 26-2.

Figure 26-1: Location Map, Gays River, Nova Scotia

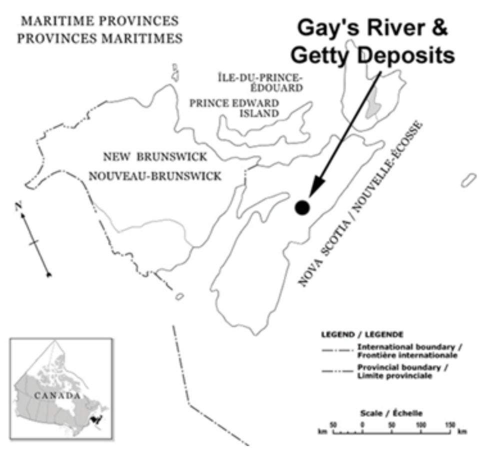
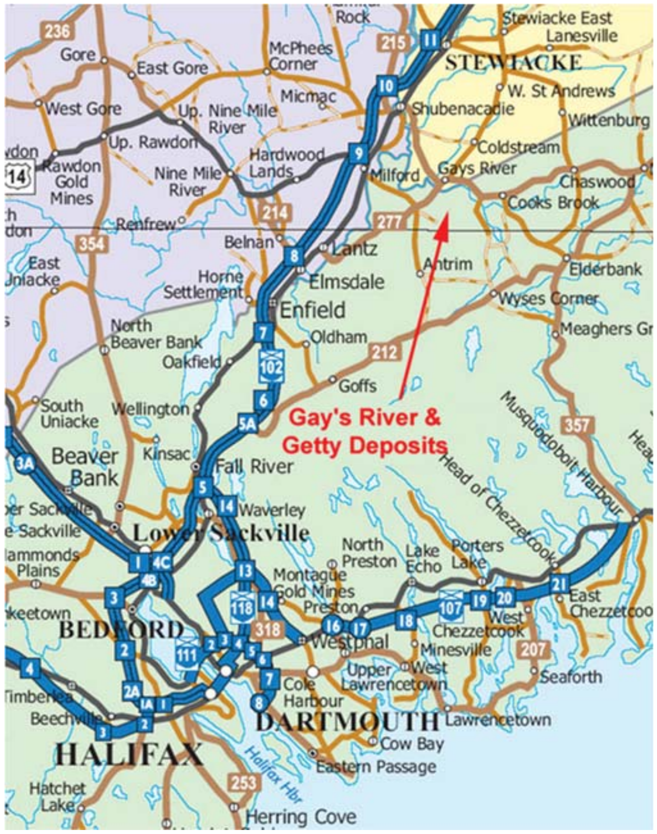


Figure 26-2: Location Relative to Halifax



26.7 Royalty Agreement

Cullen et al. (2011) described a royalty agreement that covers the Getty Deposit:

> "*Acadian advised Mercator and Selwyn that Licence 06959 that covers the Getty Deposit, plus certain peripheral claims in the area, are subject to an agreement between Acadian and Globex Resources Ltd., dated October 10th 2006, that provides Globex with a 1% Net Smelter Return (NSR) royalty interest in the associated claims plus 25,000 common shares of Acadian. Agreement terms also allow Acadian to purchase 50% of the NSR for $300,000 CDN. Mercator did not review or confirm terms of the Acadian-Globex agreement for purposes of this report and has relied upon Acadian and Selwyn for this information.*"

26.8 Mineral Lease

A Mineral Lease entirely covers (#10-1) the Scotia Mine site (Gays River Deposit). It was originally granted by the Nova Scotia Government to Westminer Canada Limited on 02 April 1990. It was originally granted as a "Mining Lease." However, changes to the Nova Scotia Mineral Resources Act that came into effect in

November 2004 changed the terminology such that existing "Mining Leases" are now known as "Mineral Leases."

The anniversary date (review date) of Mineral Lease #10-1 is 02 April of each year. Table 26-2 lists the claims comprising the Mineral Lease. Table 26-2 for the Getty Deposit shows its location. The lease conveys the rights to all minerals except coal, uranium, salt, and potash. The lease was transferred to Savage Resources in 1996 and later to Pasminco Resources Canada Company in 1999. It was finally transferred to ScoZinc in 2002. The duration of the lease is twenty years, at which time it may be renewed. The expiry date of the lease is 02 April 2030.

The Nova Scotia government held a reclamation security (bond) for the lease in the amount of $712,210. Selwyn, as a condition of the acquisition of ScoZinc in 2011, instructed its Nova Scotia counsel to pay the Nova Scotia government $1,887,790 in additional bonding for a total bond amount of $2.6 million.

As well, Selwyn instructed its Nova Scotia counsel to pay the Nova Scotia government $892,876.72 in provincial royalty payments for ScoZinc's past production.

Table 26-2: Mineral Lease 10-1 (38 Claims)

Tract Map (NTS) 11E-3B		
Tract	**Claims**	**Number of Claims**
5	NOP	3
19	JKPQ	4
20	BCDE FGK LMNO PQ	13
28	DEKL MNOP	8
29	ABCD FGH JKQ	10
Total		**38**

26.8.1 Getty Deposit

Cullen et al. (2011) described the surface or real property rights that cover the Getty Deposit:

> *"Acadian advised Mercator that surface rights to lands covering the Getty Deposit are owned under separate titles by Allan Benjamin, David Benjamin and Heather Killen. Mercator did not review the access agreements for purposes of this report but assumes that similar access permission to enter the lands for exploration purposes will be established by Selwyn. The mineral exploration claims and permits currently in place with respect to the Getty project are adequate for execution of technical programs recommended in this report. Permits necessary to do the proposed program will be applied for as required. There is adequate suitable land within the claim area for the recommend work program and future*

mining activities; however, Selwyn does not hold surface rights to this land. Selwyn will negotiate suitable purchase arrangements when the economic viability of the project has been demonstrated." Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The property is in a rural-residential area of central Nova Scotia that is typified by rolling topography and abundant surface water. The Gays River Deposit lies along the south side of the Gays River main branch, immediately east of the confluence with the Gays River south branch. The Getty Deposit lies immediately west of the Gays River Deposit, on the north side of Highway 224.

The Gays River watershed is characterized by gently rolling topography, having a maximum elevation of 170 m, an extensive cover of deciduous forest, a small population and local agricultural land development. Lakes, ponds, and rivers are sparsely distributed throughout the watershed. Typical vegetation consists of northern black spruce, balsam fir, and juniper, with birch in more wet areas. Areas of open bog occur on part of the claims. Currently, parts of the forest are being harvested or thinned.

26.9 History

26.9.1 Getty Deposit

The following is adapted from Section 5 of Cullen et al. (2011):

"… with the exception of regional soil geochemical surveying by Penarroya Ltd. in 1964 (Rabinovitch, 1967) that did not identify the Getty Deposit, no substantial mineral exploration efforts appear to have been carried out on the current Getty property prior to its acquisition by Getty in 1972.

"Exploration in the current deposit area was initiated in 1972 by Getty and joint venture partner Skelly Mining Corporation under terms of an option - purchase agreement with Millmore-Rogers Syndicate.

"Discovery of the Getty zinc-lead deposit is attributed to drill hole GGR-12 which was completed in 1972 and intersected 4.63 meters of dolomite grading 15.48% combined zinc-lead, beginning at a down hole depth of 93.11 meters. Subsequent completion of over 200 holes by Getty and Imperial on and around the property served to delineate a nearly continuous mineralized zone measuring approximately 1300 meters in length and up to 200 meters in width (Comeau, 1973, 1974; Comeau and Everett, 1975).

"Getty retained MPH Consulting Limited (MPH) to assess three development scenarios for the deposit and Riddell (1976) reported results of this work, which showed that production of 375,000 tonnes per year would be necessary to support a viable, stand-alone open pit operation.

"In 1980 economic aspects of developing the deposit based on an in-house tonnage and grade model were assessed by Esso (MacLeod, 1980). This study concluded that mining through open-pit methods as an ore supplement to the Gays River deposit would be economically viable, provided that important operating assumptions were met. The earlier MPH work was also reviewed at this time and some economic models updated. None of the work indicated that profitable stand-alone development of the deposit could be expected under market conditions of the time. George (1985) subsequently reviewed earlier evaluations and also reached a negative conclusion regarding development potential.

"In 1992 Westminer completed a resource estimate and preliminary economic assessment of the deposit based on Getty drilling results, with potential development in conjunction with the adjacent Gays River deposit being considered (Hudgins and Lamb, 1992). Results showed that milling of about 550 tonnes per day of Getty ore could be undertaken at a low cost if excess milling capacity at Gays River was being filled by such material. Westminer also indicated that zinc oxide production from the deposit would result in a substantially better financial return to the mine in comparison with a conventional smelter contract for sulphide concentrates.

"In December, 2007, Mercator completed an inferred resource estimate for the property, on behalf of Acadian, which was reported by Cullen et al.. (2007) and updated by Cullen et al.. (2008). Acadian completed a total of 138 new drill holes in support of these estimates." (Cullen et al., 2011, section 5.2)

26.10 Ownership History

26.10.1 Getty Deposit

The following is adapted from Cullen et al. (2011), Section 5.1:

"The Getty Property was acquired by Getty in 1972, at which time Getty and joint venture partner Skelly Mining Corporation began exploration under terms of an option - purchase agreement with Millmore-Rogers Syndicate.

"Claims covering the Getty Deposit were placed under closure in 1987 by the Nova Scotia government and a tender was subsequently let for acquisition of exploration rights to the property. In 1990 Westminer Canada Limited (Westminer) was deemed the successful bidder and awarded a Special Exploration License for further assessment of the deposit. Attempted renewals of the Getty Special Exploration License by Westminer for three consecutive years were not successful.

"Between 1992 and September 2006 Getty property claims were maintained under government closure and no work was carried out.

"Pasminco Resources Canada Company (Pasminco) acquired the adjacent Gays River Deposit and infrastructure in 1999 through purchase of Savage Resources Inc., and in 2000 Pasminco submitted an application to NSDNR for a Special Mining Lease covering the deposit. No lease was issued and the closed status of the property was maintained.

"In September, 2006 the provincial government tendered exploration rights to the closed Getty property and Exploration Licenses 6959 and 6960 were subsequently issued to Acadian on October 20th, 2006 as successful bidder under the tendering process. During the 2007-2008 period, Acadian carried out a substantial amount of diamond drilling in the deposit area and prepared two National Instrument 43-101 compliant mineral resource estimates.

"In February 2011, Selwyn Resources Limited ("Selwyn") purchased ScoZinc and related zinc-lead exploration properties, including the Getty deposit exploration licenses."

26.11 Historical Mineral Resource and Mineral Reserve Estimates

The following resource and reserve estimates are historical in nature, have not been extensively audited by the authors, were not prepared according to NI 43-101 (except where noted) and should not be relied upon.

26.11.1 Getty Deposit

The following is taken from Cullen et al. (2011):

"Four previous estimates of tonnage and grade for in-situ mineralization comprising the Getty Deposit are available in the public record. The earliest of these was prepared for Getty by MPH Consulting Limited (Riddell, 1976) and was revised in 1980 as part of a Mine Valuation Study carried out for Esso (MacLeod, 1980). Subsequently, Westminer developed an in-house estimate and preliminary economic assessment of the deposit based on historic drilling (Hudgins and

Lamb, 1992). The fourth estimate was completed in December, 2007 by Mercator for Acadian and reported by Cullen et al (2007).

"Results of the first three historic estimates are presented below in Table 4a and all pertain to areas currently covered by Acadian exploration licences. These pre-date National Instrument 43-101 (NI 43-101) and have not been classified under Canadian Institute of Mining, Metallurgy and Petroleum Standards for Reporting of Mineral Resources and Reserves: Definitions and Guidelines (the CIM standards). On this basis they should not be relied upon. Table 4b presents the Cullen et al. (2007) NI43-101 compliant resource estimate completed by Mercator, which has an effective date of December 12th, 2007.

Table 26-3: Historic Resource Estimates for Getty Deposit Not NI 43-101 Compliant (from Cullen et al., 2011)

Reference	Tonnes	Zn + Pb %	Zn %	Pb %
Riddell(1976)	4,470,400	3.71	1.87	1.84
MacLeod(1980)	3,149,600	2.97	1.60	1.37
Hudgins and Lamb(1992)	4,490,000	3.20	1.87	1.33

Notes: With regard to the historic mineral resource estimates stated above 1) a qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves; 2) the issuer is not treating the historical estimate as current mineral resources or mineral reserves as defined in sections 1.2 and 1.3 of NI 43-101; and 3) the historical estimate should not be relied upon.

Table 26-4: Mercator NI 43-101 Compliant Resource Estimate for Getty Deposit *(2007) (from Cullen et al., 2011)*

Resource Category	Zn % + Pb % Threshold	Tonnes (Rounded)	Pb %	Zn %	Zn % + Pb %
Inferred	2.00	4,160,000	1.40	1.81	3.21

"Riddell(1976) used a 2% (zinc% + lead%) cut-off, Macleod (1980) used 1.5% zinc cut-off and Hudgins and Lamb (1992) used a 1.5% zinc-equivalent cut-off defined as zinc equivalent = zinc% +(lead % x 0.60). Figures for the previous Mercator estimate that are presented in Table 4b reflect application of a 2% zinc + lead cut-off. The Riddell (1976) and MacLeod (1980) estimates are based on drill-hole-centered polygonal methods of volume estimation along with subjectively determined specific gravity factors reflecting general experience. Both estimates include length-weighted drill hole grade assignments to polygons with subsequent tonnage-weighting to determine deposit grades. In contrast, Hudgins and Lamb (1992) used Surpac® deposit modeling software, a cross sectional method of volume estimation, a single assigned specific gravity factor of 2.75 g/cm3 and calculated average deposit zinc and lead grades as the length-weighted averages of all qualifying drill hole intercepts. Further discussion of

historic resource estimates plus that by Mercator appears in report section 16.4." (Cullen et al, 2011, section 5.3)

26.12 Mineralization

26.12.1 Getty Deposit

The following is taken from section 8 (Mineralization) of Cullen et al (2011):

"Zinc and lead sulphide mineralization are found throughout the Getty carbonate bank, along with trace amounts of iron sulphide in isolated areas. Base metal sulphides are also present to a lesser extent in carbonate matrix of the underlying conglomerate/breccia unit and within calcite or micrite filled fractures and joints present in underlying Goldenville Formation greywackes. While not extensively reported to date, galena has also been documented locally at the Scotia Mine deposit in thin (<20cm thick) discordant, steeply dipping veins that generally trend north-south (B. Mitchell, personal communication, 2007)

"Drilling to date on the Getty deposit has shown that massive to submassive high grade mineralization like that commonly present along steep bank front zones at Scotia Mine is not present to a significant degree at Getty (Bryant, 1975). However, a clear association of higher zinc and lead grades with dolostone intervals on the northeast and north slopes of the Getty bank is recognized and lower grades over thicker intervals occur within the carbonate sections at the top of the bank. Mineralization is more poorly developed along the southwest side of the bank.

"Sphalerite is the predominant base metal sulphide phase present and is typically honey yellow to buff or beige in color and finely crystalline. Based on drill core observations, Bryant (1975) specified the following four modes of sphalerite occurrence within the deposit, with the first being the most common: (a) disseminated mineralization showing concentrations from trace to 10% or more, (b) semi-massive and massive mineralization as seams and replacements along bedding surfaces or laminae, (c) massive, porosity filling or surface coating mineralization in fossiliferous and vuggy carbonate, (d) mineralization associated with secondary calcite in small stringers and veinlets.

"Silver is a trace constituent of the Getty sulphide assemblage but is not present at levels of economic significance. This parallels the situation at adjacent Scotia Mine where Roy et al (2006) reported historic silver values in mill concentrates that were typically less than 40 parts per million." (Cullen et al, 2011, section 8).

26.13 Getty Deposit Exploration

A description of mineral exploration work that was carried out on the Getty Deposit was given in Cullen et al. (2011):

"… with the exception of regional soil geochemical surveying by Penarroya Ltd. in 1964 (Rabinovitch, 1967) that did not identify the Getty Deposit, no substantial mineral exploration efforts appear to have been carried out on the current Getty property prior to its acquisition by Getty in 1972.

"Exploration in the current deposit area was initiated in 1972 by Getty and joint venture partner Skelly Mining Corporation under terms of an option - purchase agreement with Millmore-Rogers Syndicate.

"Discovery of the Getty zinc-lead deposit is attributed to drill hole GGR-12 which was completed in 1972 and intersected 4.63 meters of dolomite grading 15.48% combined zinc-lead, beginning at a down hole depth of 93.11 meters. Subsequent completion of over 200 holes by Getty and Imperial on and around the property served to delineate a nearly continuous mineralized zone measuring approximately 1300 meters in length and up to 200 meters in width (Comeau, 1973, 1974; Comeau and Everett, 1975).

"Mercator completed a National Instrument 43-101 compliant Inferred Mineral Resource Estimate for Acadian on the Getty Deposit with an effective date of December 12, 2007. This initial estimate was subsequently updated in a new National Instrument 43-101 compliant resource in 2008 (Cullen et al., 2008) after a total of 10,620 meters of drilling in 138 diamond drill holes had been completed by Acadian on the Getty property under the direct supervision of Mercator staff. The information used to complete these estimates was compiled from the 2007-2008 drilling by Acadian plus historical drilling undertaken prior to Acadian's involvement in the property.

"Acadian initiated a major diamond drilling program on the Getty property in July 2007, and Mercator provided all site supervision, logging, sampling and quality control/quality assurance services to Acadian for this program, which consisted of 138 diamond drill holes. The purpose of the drilling was to upgrade geological confidence in the deposit, provide a basis for the new mineral resource estimate and to provide a higher category classification to the mineral resource estimate (Cullen et al, 2008)."

26.14 Drilling

26.14.1 Sample Length – True Width Relationship

The sample intervals do not necessarily represent true widths. The orientation of the deposit is variable, meaning the true width of any given intercept must be calculated with reference to the geological model. The orientation of the deposit is well known and is described in Section 7.2.

26.14.2 Getty Deposit Drilling

Drilling on the Getty Deposit was described in Cullen et al. (2011).

Historic diamond drilling information pertaining to the Getty deposit was compiled by Westminer in a digital database containing information for approximately 181 vertical holes totaling 16,875 meters of drilling. The Westminer database was originally prepared to support the resource estimate reported by Hudgins and Lamb (1992) and to this end, collar coordinates, lithologic codes, geologic legend and individual drill core assay interval results were compiled from original drill logs, checked for errors, and entered into the original digital database. All historic holes were initially coordinated to local Getty reference grid but Mercator subsequently transformed all drill hole coordinates into the Scotia Mine grid using historic tie points for which Acadian surveyors provided up to date mine grid coordination. Universal Transverse Mercator (UTM) coordinates (Zone 20, NAD 83 Datum) were also calculated by Mercator for all holes in the project database and a listing of drill hole coordinates and orientation data for the deposit in the block model grid system appears in Cullen et al. (2011). Mercator staff physically checked all drill hole entries in the database against the original hard copy logs.

Between July 2007 and April 2008, Acadian completed 10,620 meters of drilling in 138 diamond drill holes on the Getty property under the direct supervision of Mercator staff. The drilling program focused on 1) validation of past drilling results, 2) infilling in areas where insufficient information existed to define mineral resources or in areas where upgrading of existing Inferred mineral resources to Indicated or Measured categories was possible, 3) re-drilling of historic holes where information on sampling and assays were missing and 4) extension of mineralized zone limits beyond those previously defined. Table 10-4 below present's collar information for all drill holes completed by Acadian during the 2007-2008 program and a drill collar location plan is included in Cullen et al. (2011).

Table 26-5: Collar Information by Acadian during 2007-2008

Hole Number	Collar Coordinates Easting (m)	Collar Coordinates Northing (m)	Collar Elevation (m)	Angle (Deg.)	Depth (m)
S992-07	6893.91	6584.66	556.35	-90	74
S993-07	6929.3	6618.11	549.27	-90	94
S994-07	6856.41	6554.85	557.89	-90	80
S995-07	6821.7	6521.68	555.54	-90	80
S996-07	6848.41	6595.56	556.96	-90	71
S997-07	6930.65	6508.78	556.57	-90	77
S998-07	6781.44	6593.48	557.26	-90	56
S999-07	6814.47	6624.98	555.77	-90	59
S1000-07	6885.79	6686.53	545.67	-90	86
S1001-07	6845.17	6658.82	549.86	-90	68
S1002-07	6786.06	6490.32	553.46	-90	61
S1003-07	6768.48	6552.21	555.35	-90	55
S1004-07	6752.62	6685.67	552.13	-90	70
S1005-07	6721.44	6644.97	554.69	-90	62
S1006-07	6686.72	6625.8	557.38	-90	47
S1007-07	6677.94	6663.15	555.31	-90	59
S1008-07	6683.03	6555.39	561.89	-90	41
S1009-07	6660.73	6593.76	562.19	-90	44
S1010-07	6614.98	6667.11	558.38	-90	50
S1011-07	6659.09	6707.74	551.81	-90	62
S1012-07	6682.36	6743.91	548.51	-90	73
S1013-07	6565.82	6741.08	553.64	-90	41
S1014-07	6578.49	6782.15	548.88	-90	44
S1015-07	6548.12	6785.4	548.88	-90	35
S1016-07	6535.89	6832.24	545.65	-90	35
S1017-07	6617.88	6791.98	549.48	-90	47
S1018-07	6609.9	6750.15	551.23	-90	44
S1019-07	6716.98	6769.9	548.11	-90	41
S1020-07	6685.49	6840.5	545.4	-90	92
S1021-07	6731.32	6614.94	555.01	-90	88
S1022-07	6720.95	6531.89	558.79	-90	38
S1023-07	6681.89	6793.99	548.38	-90	89
S1024-07	6726.11	6814.06	547.27	-90	116
S1025-07	6651.11	6897.88	541.49	-90	62
S1026-07	6622.5	6932.19	539.15	-90	71
S1027-07	6597.45	6897.27	541.61	-90	56
S1028-07	6627.4	6863.73	543.79	-90	62

Hole Number	Collar Coordinates Easting (m)	Collar Coordinates Northing (m)	Collar Elevation (m)	Angle (Deg.)	Depth (m)
S1029-07	6695.96	6898.51	542.29	-90	82
S1030-07	6565.23	6857.1	544.22	-90	53
S1031-07	6749.13	6795.28	547.15	-90	110
S1032-07	6546.38	6900.8	540.64	-90	46
S1033-07	6654.41	6851.35	544.55	-90	66
S1034-07	6774.93	6837.9	544.93	-90	121
S1035-07	6721.65	6897.27	542.28	-90	109
S1036-07	6805.61	6879.92	543.86	-90	146
S1037-07	6751.97	6930.99	539.29	-90	107
S1038-07	6772.03	6951.67	537.47	-90	104
S1039-07	6603.08	7032.43	533.53	-90	78
S1040-07	6794.17	6918.98	541.32	-90	137
S1041-07	6670.44	6962.86	537.76	-90	61
S1042-07	6673.35	7031.89	529.53	-90	62
S1043-07	6744.07	6997.2	531.85	-90	80
S1044-07	6964.92	6534.27	554.71	-90	68
S1045-07	6993.7	6571.8	549.4	-90	80
S1046-07	6728.98	7029.9	527.17	-90	89
S1047-07	7033.36	6544.93	548.43	-90	62
S1048-07	7070.89	6511.28	547.12	-90	89
S1049-07	6698.73	6997.98	531.93	-90	60
S1050-07	7036	6590.68	545.38	-90	95
S1051-07	6731.34	6719.44	550.07	-90	76
S1052-07	6864.45	6441.26	552.43	-90	92
S1053-07	6857.46	6523.76	557.03	-90	89
S1054-07	6913.68	6433.7	553.87	-90	116
S1055-07	6999.86	6326.58	546.89	-90	151
S1056-07	6952.4	6314.51	544.55	-90	83
S1057-07	6975.89	6618.16	546.16	-90	101
S1058-08	6925.51	6667.45	545.26	-90	101
S1059-08	6997.46	6512.28	553.1	-90	71
S1060-07	7032.31	6419.19	548.38	-90	96
S1061-08	7005.67	6381.08	550.01	-90	121
S1062-08	7103.53	6470.53	544.62	-90	92
S1063-08	6795.97	6801.93	546.91	-90	107
S1064-08	6898.19	6224.86	535.36	-90	43
S1065-08	6853.43	6228.9	537.35	-90	64
S1066-08	6883.95	6318.98	540.69	-90	60

Hole Number	Collar Coordinates Easting (m)	Collar Coordinates Northing (m)	Collar Elevation (m)	Angle (Deg.)	Depth (m)
S1067-08	6883.85	6114.43	538.11	-90	48
S1068-08	6917.47	6721.67	544.39	-90	113
S1069-08	6906.4	6055.05	532.45	-90	71
S1070-08	6908.98	6368.17	548.36	-90	92
S1071-08	6826.52	6747.33	549.09	-90	88
S1072-08	6851.27	6786.47	546.98	-90	95
S1073-08	6742.15	6144.4	530.59	-90	27
S1074-08	6607.02	6107.87	524.85	-90	78
S1075-08	6947.63	6763.13	542.18	-90	113
S1076-08	6672.95	6163.98	528.14	-90	43
S1077-08	6811.59	6037.49	526.52	-90	60
S1078-08	6533.73	6301.72	543.98	-90	83
S1079-08	6549.57	7039.43	528.97	-90	80
S1080-08	6584.42	6310.65	543.13	-90	68
S1081-08	6637.48	6293.16	538.46	-90	32
S1082-08	6561.66	7000.93	531.65	-90	77
S1083-08	6616.27	6221.05	529.77	-90	44
S1084-08	6500.18	6997.6	530.26	-90	101
S1085-08	6526.73	6184.7	528.75	-90	117
S1086-08	6482.64	6911.99	538.37	-90	76
S1087-08	6468.44	6959.88	533.3	-90	95
S1088-08	6538.84	6228.15	532.35	-90	51
S1089-08	6535.1	6226.14	532.34	-90	95
S1090-08	6537.21	6351.61	552.64	-90	86
S1091-08	6728.05	7111.32	513.44	-90	83
S1092-08	6604.71	6354.78	550.65	-90	80
S1093-08	6763.48	7081.3	514.8	-90	59
S1094-08	6521.74	6394.32	560.35	-90	86
S1095-08	6803.17	7074.47	512.95	-90	72
S1096-08	6478.63	6368.11	556.93	-90	104
S1097-08	6434.31	6928.53	533.35	-90	95
S1098-08	6538.88	6442.45	564.27	-90	68
S1099-08	6468.35	6412.56	561.4	-90	112
S1100-08	6472.48	6865	539.96	-90	57
S1101-08	6512.89	7035.97	529.15	-90	104
S1102-08	6551.6	6494.31	564.66	-90	71
S1103-08	6594.06	6495.9	565.4	-90	62
S1104-08	6440.43	6964.51	531.46	-90	50

Hole Number	Collar Coordinates Easting (m)	Collar Coordinates Northing (m)	Collar Elevation (m)	Angle (Deg.)	Depth (m)
S1105-08	6557.47	6553.06	564.25	-90	122
S1106-08	6610.53	6545.52	566.28	-90	62
S1107-08	6518.11	6592.52	562.51	-90	101
S1108-08	6422.59	6869.34	539.52	-90	58
S1109-08	6467.99	6317.44	549.34	-90	59
S1110-08	6369.43	6864.19	537.5	-90	41
S1111-08	6389.85	6915.57	533.73	-90	73
S1112-08	6668.56	6196.65	528.77	-90	30
S1113-08	6656.95	6116.13	526.39	-90	78
S1114-08	6281.48	6867.3	535.48	-90	23
S1115-08	6662.51	6243.31	531.36	-90	26
S1116-08	6570.75	6406.75	561.01	-90	68
S1117-08	6257.15	6957.27	528.04	-90	62
S1118-08	6490.24	6252.22	539.16	-90	104
S1119-08	6551.7	6113.51	523.32	-90	77
S1120-08	6314.82	6982.61	527.53	-90	36
S1121-08	6236.26	6900.8	529.95	-90	38
S1122-08	6571.44	6219.21	530.35	-90	75
S1123-08	6960.44	6664.49	544.9	-90	116
S1124-08	6896.65	6538.24	557.69	-90	80
S1125-08	6987.14	6469.86	553.76	-90	89
S1126-08	6817.12	6150.5	534.95	-90	38
S1127-08	6489.42	6146.45	529.21	-90	137
S1128-08	6851.05	5283.15	543.74	-90	218
S1129-08	6256.73	6257.29	555.39	-90	177

[a] Data supplied by ScoZinc.

The complete Getty project drilling database includes results of the 138 diamond drill holes recently drilled by Acadian and the 184 historic drill holes completed during the 1970s, 181 of which were drilled by Getty and total 16,875 m of drilling. The three remaining holes were completed by Esso during the same time period and totaled 157 m of drilling. The resource outline pertinent to this report includes all of the 138 Acadian holes and 68 of the historic drill holes.

All holes were drilled vertically and mineralized intercepts from holes drilled on the bank top, where mineralization is generally horizontal, represent true width. Mineralization intercepts from holes drilled on the bank front, where mineralization slopes, have a true width that is 60-70% of the intercept width. Drill hole core recovery for Acadian drilling was in excess of 90% and recovery was not a factor in the resource estimation. A review of logs for historic drill holes and re-logging of select historic holes by Mercator did not identify core loss as an issue.

26.14.3 Logistics of Acadian Drill Program

Logan Drilling of Stewiacke, Nova Scotia, was contracted to complete 2007-2008 drilling utilizing skid mounted Longyear 38 drilling equipment equipped to recover NQ sized drill core (4.76 cm diameter). One drill was typically employed, but a second drill was periodically on site. Both machines typically operated on a 24 hour per day basis. Mercator was contracted to manage day to day drilling operations and provided onsite supervision, transportation of core to the secure logging facility at Acadian's Scotia Mine, plus logging of drill core and supervision of core sampling services. A registered land surveyor surveyed drill hole collars, and all drill holes were coordinated to the local Scotia Mine grid system.

26.15 Sample Preparation, Analysis and Security

26.15.1 Getty Deposits (2008)

26.15.1.1 Site Procedures

Cullen (2011) provided the following description for the sampling methods that were used for the 2008 drilling program (Gays River and Getty deposits).

26.15.1.2 Sample Security and Chain of Custody

In accordance with the sample protocol established by Mercator for the 2008 drilling program, all drill core was delivered from the drill site to the secure and private core logging facility at Acadian's Scotia Mine by either Logan Drilling Limited staff or Mercator field staff. Drill core logging was carried out by a Mercator geologist who also marked core for sampling and supervised core splitting by a technician using a rock saw. Sample tag numbers from a three-tag sample book system were used for the program, with one tag showing corresponding down hole sample interval information placed in the sampled core boxes at appropriate locations, one tag lacking down hole interval information placed in the core sample bag for shipment to the laboratory, and the third tag with sample interval information retained in the master sample book for future reference and database entry purposes. After sampling, core boxes were closed and placed in storage at the Scotia Mine site. Sealed sample bags were placed in an ordered sequence prior to insertion of quality control samples, preparation of sample shipment documentation, checking, and placement in plastic buckets for shipment by commercial courier to Eastern Analytical Limited ("Eastern"), a recognized commercial laboratory located in Springdale, Newfoundland. A check pulp sample split was prepared at Eastern for every 25th submitted sample and these were labelled, placed in a sealed envelope, and returned to Mercator. After insertion of certified standard and blank samples, all check samples were sent to ALS Chemex in Sudbury, Ontario, for independent analysis of Zn and Pb levels. All other prepared pulps and coarse reject material was stored at Eastern until the end of the program, at which time they were shipped back to Scotia Mine for secure archival storage.

26.15.2 Laboratory Procedures

Cullen (2011) provided the following description for the sampling methods that were used for the 2008 drilling program (Gays River and Getty deposits).

26.15.2.1 Core Sample Preparation

Core samples received by Eastern were organized and labelled and then placed in drying ovens until completely dry. Dried samples were crushed in a Rhino Jaw Crusher to consist of approximately 75% minus 10 Mesh material. The crushed sample was riffle split until 250 grams to 300 grams of material was separated and the remainder of the sample was bagged and stored as coarse reject. The 250 gram to 300 gram split was pulverized using a ring mill to consist of approximately 98% minus 150 Mesh material. All samples underwent ICP analysis, for which a 0.50 g portion of the pulverized material was required. Those samples containing greater than 2,200 ppm of Zn or Pb were then processed using ore grade analysis for which 0.20 g of pulverized material was required. Laboratory sample preparation equipment was thoroughly cleaned between samples in accordance with standard laboratory practice.

Check sample splits of pulverized core were submitted to the ALS Chemex laboratory facility in Sudbury, Ontario, as part of the project quality control and assurance protocol. This material was prepared in approximately 100 g bagged splits by Eastern and returned to Mercator for subsequent submission to ALS Chemex. Since the received split material had already been pulverized, further preparation was limited to homogenization and splitting of a 0.4 g portion for subsequent analysis.

26.15.2.2 Core Sample Analysis

Eastern Analytical procedures outlined below pertain to all core samples from the 2008 drill program.

ICP Analysis: A 0.50 g sample is digested with 2 ml HNO_3 in a 95 °C water bath for ½ hour, after which 1 ml HCl is added and the sample is returned to the water bath for an additional ½ hour. After cooling, samples are diluted to 10 ml with de-ionized water, stirred, and let stand for 1 hour to allow precipitate to settle.

For ore grade analysis base metals (Pb, Zn, Cu), a 0.20 g sample is digested in a beaker with 10 ml of nitric acid and 5 ml of hydrochloric acid for 45 minutes. Samples are then transferred to 100 ml volumetric flasks and analyzed on the Atomic Absorption Spectro-Photometer (AA). The lower detection limit is 0.01% and the upper detection limit is greater than 2,200 ppm Pb or Zn.

For silver, a 1000 mg sample is digested in a 500 ml beaker with 10 ml of hydrochloric acid and 10 ml of nitric acid with the cover left on for 1 hour. Covers are then removed and the liquid is allowed to evaporate leaving a moist paste. Following this, 25 ml of hydrochloric acid and 25 ml of deionised water are added and the solution is gently heated and swirled to dissolve the solids. The cooled material is transferred to 100 ml volumetric flask and is analyzed using AA. The lower detection limit is 0.01 oz/t of silver with no upper detection limit.

A prepared sample is digested in 75% aqua regia for 120 minutes. After cooling, the resulting solution is diluted to volume (100 ml) with de-ionized water, mixed and then analyzed by inductively coupled plasma - atomic emission spectrometry or by atomic absorption spectrometry.

26.16 Data Verification

26.16.1 Getty Deposit

Data verification measures for the Getty Deposit were described in Cullen et al. (2011):

> *"Review by Mercator of all government assessment reports and internal Acadian files available from the Scotia Mine site established that typed lithologic logs with complete assay records from the Getty drilling era were available. However, original sample record books, laboratory reports and other associated information were not found. The digital drill hole database used for the Westminer's 1992 resource estimate was also obtained from Acadian and validated against the original hard copy drill log and assay record entries. Checking of digital records included manual inspection of individual database lithocode entries against source hard copy drill logs as well as use of automated validation routines that detect specific data entry logical errors associated with sample records, drill hole lithocode intervals, collar tables and down-hole survey tables. Drill hole intervals were also checked for sample interval and assay value validity against the original drill logs. Database entries were found to be of consistently acceptable quality but minor lithocode and assay entry corrections were made by Mercator. These were incorporated to create the validated and functional drilling database used in the resource estimate. As noted earlier, original assays certificates were not found for any of the historic drilling programs and no records of the laboratories to which samples were submitted for analysis,*

or methods of analysis, were documented in any of the historic drilling reports reviewed for the resource estimate.

"As part of the validation process, Mercator staff visited the NSDNR Core Library in Stellarton, Nova Scotia to review and sample core from the archived Getty drill holes. Nineteen holes where examined but only one hole GGR-212 was re-logged in detail and ten holes … were re-sampled and analyzed for purposes of quality control and quality assurance. These provided additional verification of historical assays and logging results. Results of this and related programs are presented below under separate headings." (Cullen et al., 2011, section 13.1)

"Combined results of the Getty drill hole re-sampling and twin hole programs by Acadian generally support the earlier conclusion of Cullen et al.. (2008), based on a smaller data set, that validated historic drilling information represented in Acadian's Getty deposit database is of acceptable quality for resource estimation purposes." (Cullen et al..).

26.17 Mineral Resource Estimate

The Gays River Deposit's mineral resource estimate was prepared by Doug Roy, P. Eng. of MineTech International Limited and Mr. Tim Carew, P. Geo. of Reserva International LLC. Getty's mineral resource estimate was prepared by Cullen et al. (2011) of Mercator Geological Services. The estimates were separately prepared using slightly different parameters, the most significant of which were different Zn-equivalent grade formulae and different block cutoff grades for resource reporting. These differences preclude reporting a total for both deposits. In other words, mineral resources for the Gays River and Getty Deposits are reported separately.

Only Mineral Resources were identified. No economics work, such as estimating capital and operating costs, that would be required for identifying Mineral Reserves, was carried out. In other words, no Mineral Reserves were identified.

26.17.1 Getty Deposit

Cullen et al. (2011) estimated the Getty Deposit's mineral resources. The following (i.e., the entirety of Section 26.17) is an excerpt from that report.

"The definition of mineral resource and associated mineral resource categories used in this report are those recognized under National Instrument 43-101 and set out in the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves Definitions and Guidelines (the CIMM Standards).

26.17.2 Geological Interpretation Used in Resource Estimation

"All areas of zinc-lead mineralization included in the current resource are restricted to the Getty Deposit carbonate bank and occur within dolomitized Gays River Formation lithologies. For resource model purposes the Getty Deposit is considered an extension of the adjacent Gays River Deposit and both are classified as carbonate-hosted, stratabound zinc-lead deposits of the Mississippi Valley Type (MVT). Mineralization is localized in carbonate bank lithofacies that developed above and around paleo-topographic basement highs comprised of Cambro-Ordovician Goldenville Formation greywacke and slate. By definition, Gays River Formation lithologies are laterally equivalent to laminated and thin bedded limestones of the Macumber Formation.

"Zinc and lead mineralization of economic proportions is exclusively developed within dolomitized carbonate bank lithologies at Getty and is considered directly comparable to that seen on the adjacent Scotia Mine property. Sphalerite and galena are the dominant sulphide minerals present but trace amounts of marcasite/pyrite occur locally, typically as cavity-lining phases that post-date the zinc-lead mineralizing stage. Silver does not occur in economic proportions in this district but does report to Scotia Mine concentrates at levels of about one ounce per tonne. A similar presence at Getty may exist. Barite is absent from the deposit, as is celestite, but traces of fluorite have been reported (Kontak, 1998, 2000; Sangster et. al., 1998).

"As noted earlier, several types of lead and zinc mineralization are represented in the related Scotia Mine and Getty Deposits, the most important of which are (1) submassive to massive replacements of carbonate bank lithofacies by sphalerite and galena, typically along steeply dipping carbonate bank front intervals that face the open paleo-basin, (2) disseminated, replacement and porosity filling phases within various carbonate bank lithologies adjacent to and within bank-front intervals, and (3) in rare vein and irregular vug settings or as matrix mineralization between greywacke clasts or boulders in a basal breccia unit that typically separates carbonate bank lithologies from basement greywacke. The dominant type of mineralization in the Getty Deposit is disseminated in nature.

26.17.3 Methodology of Resource Estimation

26.17.3.1 Overview of 2011 Estimation Procedure

"The Getty mineral resource estimate is based on a three-dimensional block model developed using Surpac Version 6.0.3 modeling software and the validated project drill hole database. The database includes results from 181 historic diamond drill holes completed by Getty as well as 4 holes completed by

Esso and 138 diamond drill holes completed by Acadian in 2007-2008. The current resource outline includes 84 historic holes and 94 Acadian holes, although additional holes from both sources occur adjacent to the outline and were used for geological and block model peripheral constraint definition purposes.

"The first step in development of the resource model was creation of a set of interpreted geological cross sections presenting lithocoded rock types interpreted from drill logs as well as lead and zinc core sample assay interval data. These served to establish an understanding of carbonate bank geometry and grade distribution trends present in the deposit and were later augmented by contour plans depicting overburden depth, dolomite thickness and basement surface configurations. Sections were created using the local project grid at a nominal spacing of 50 meters, with adjustment of this spacing made as necessary to provide complete coverage of the deposit. Geological and grade distribution models developed from the sections were used to guide and assess subsequently developed versions of the three-dimensional block model.

"Assay results from the validated project database were initially assessed through calculation of distribution statistics for both zinc and lead populations after compositing to a common 1.0 meter support base. In total, 1672 composites were created from analytical results for 1794 original core samples. Frequency distribution and probability plots for the composite data set were also prepared and results were interpreted as showing that the few high grade samples present were reflections of valid mineralization styles for which block-scale correlations could be reasonably expected. This assertion reflects observations made during underground mining of high grade portions of the adjacent Gays River Deposit. Composites showing high zinc and lead grades occur in several areas along the north-facing bank front of the Getty Deposit, as is the case at Scotia Mine, but these are typically lower in grade, thinner and spatially less extensive than similar high grade areas at Scotia Mine. On the basis of combined factors, no requirement for high grade capping of assay results in the Getty data set was established.

"The Getty Deposit model was developed within a three-dimensional, peripheral constraint (or solid) created in Gemcom Surpac Version 6.0.3 initially based on a combination of two contributing parameters, these being (1) a minimum grade % (zinc plus lead) value of 1.00% with a minimum down-hole intercept length of 3.0 meters, and (2) lateral limits to the deposit solid defined on the basis of midpoints between mineralized and non-mineralized drill holes or a maximum 25 meter projection from a mineralized hole where no other constraining hole was present. The grade cut-off was assigned as a reflection of the deposit's near-surface location and associated potential for open pit development.

"While not as complex as that at Scotia Mine, the carbonate bank front configuration at Getty is irregular and the solid developed for deposit modelling purposes is characterised by numerous promontories and re-entrants. This is particularly true along north-facing bank front intervals that show spatial association with areas of best zinc and lead mineralization. This configuration approximates a series of variably-oriented panels of dipping mineralization that, although correlative, show strike and dip changes along the length of the deposit. The current peripheral deposit constraint solid for the block model reflects this variation and is based on that developed for the earlier Acadian resource estimate (Cullen et al., 2007). However, it differs from the earlier constraint by accommodating the new drill holes by Acadian and being comprised of 26 sub-domains reflecting areas of common mineralized zone orientation. As detailed later in this report, block grade interpolation was separately carried out in each sub-domain using unique search ellipse orientations.

"Spatial variability of mineralized zone trends at Getty prevented development of experimental variograms for the lead and zinc data set that reflected continuity of the mineralized zone to the degree seen in the original geological cross section model. This issue was addressed by Roy et al. (2006) at the Gays River Deposit through three-dimensional transformation of their deposit model that "unfolded" the various mineralized segments to a common surface. Transformed data supported acceptable variogram models and these were subsequently used to establish parameters for grade interpolation into their block model.

"In contrast to the method used at Scotia Mine, mineralized trend variability along the Getty Deposit was addressed in the current model through development of the 26 orientation domain solids within which grade interpolation was constrained. Composite populations within individual domains typically did not provide an adequate number of sample pairs to create well developed experimental variograms. However, useful variogram models for the largest northwest trending sub-domain were initially developed and these were augmented by variogram models calculated for the entire composite population occurring within the peripheral deposit constraint. In the latter case it was recognized that geometric aspects of the deposit could factor negatively in the evaluative process. Based on combined results of the two approaches, the strike and dip directions of the mineralized zones were determined to show the highest degrees of grade correlation at longest range values. This directly supported earlier qualitative geological assessment of the grade trends. Geometric aspects of the mineralized zones were used in conjunction with variogram results to select interpolation ellipse axial ranges, with common ranges used in all sub-domains in conjunction with unique assigned orientation parameters. Block grades were

assigned to the 26 deposit sub-domains using inverse distance squared (ID2) interpolation methodology.

"Results of the grade interpolation process were initially checked against geological cross sections to assess conformity and to provide primary validation of the final deposit block model. A further check on the resource model was completed using Nearest Neighbour grade interpolation methodology on the deposit solid. Resource figures reflecting ID2 interpolation and a range of minimum grade cutoff values, beginning at 2.0% (zinc + lead), constitute the final resource estimate documented in this report.

26.17.3.2 Capping of High Grade Assay Values

"Zinc and lead grades for all drill core samples were reviewed and descriptive statistics calculated for both the raw data set and that reflecting 1 meter composite support. The latter are presented below in Table 14-11 and include only those holes that intercept the deposit solid.

Table 26-6: Descriptive Statistics: 1 Meter Drill Core Composites In Resource Solid

Parameter	Zinc	Lead
Mean	1.46%	1.00%
Variance	1.94	2.53
Standard Deviation	1.39	1.59
Coefficient of Variation	0.948	1.580
Maximum	11.30	18.54
Minimum	0.00	0.00
Number	1961	1961

"Maximum zinc and lead grades at 1 meter composite support are 11.30% and 18.54% respectively and reflect zones of higher grade mineralization that are considered spatially coherent and correlative at block scale within the deposit. These form a meaningful part of the grade distribution spectrum of the deposit and are associated with valid geological domains. On this basis, high grade lead and zinc values were not capped for use in the current resource estimate.

26.17.3.3 Compositing of Drill Hole Data

"One meter down-hole composites of raw core sample assay values were created for each drill hole, with this length representing the dominant sample interval used by Acadian in the 2007-2008 drilling program. Historic drilling program sample length statistics for all holes are presented in Table 14-2. A review of associated rank and percentile figures shows that 99 percent of the historic samples measure less than 2.0 meters in length, 75 percent measure 1.52 meters or less in length and 39 percent measure less than 1.0 meter in length. Average length of historic samples is 1.15 meters.

Table 26-7: Core Sample Length Descriptive Statistics

Parameter	Historic Core Sample Length (m)	Acadian Core Sample Length (m)
Mean	1.15	1.00
Variance	0.222	0.063
Standard Deviation	0.47	0.25
Coefficient of Variation	0.411	0.250
Maximum	4.26	6
Minimum	0.02	0.38
Number	855	939

"With respect to Acadian sampling, associated rank and percentile figures show that 95 percent of samples measure 1.0 meter or less in length and 99% of samples measure 2.0 meters or less in length in length. Average length of Acadian core samples is 1.00 meters. Sampling of high grade intervals in historic drill holes was typically carried out based on geological contacts with no minimum sample length parameters applied. This may in part be reflected in samples from historic programs with lengths of less than 0.5 meters.

"In total, 1672 assay composites at 1.0 meter support were calculated within the resource estimation solids from the combined historic drill hole and Acadian drill hole data set.

26.17.3.4 Calculation of Equivalent Zinc

"The previous Mercator resource estimate for the Getty Deposit reported by Cullen et al. (2008) presented a zinc equivalent parameter of zinc equivalent = (zinc% + lead %). Riddell (1976) also used a zinc% + lead% factor to define resource cutoff values and included the parameter in the associated resource estimate. Use of zinc% + lead% to define cutoff values was not retained for the current estimate.

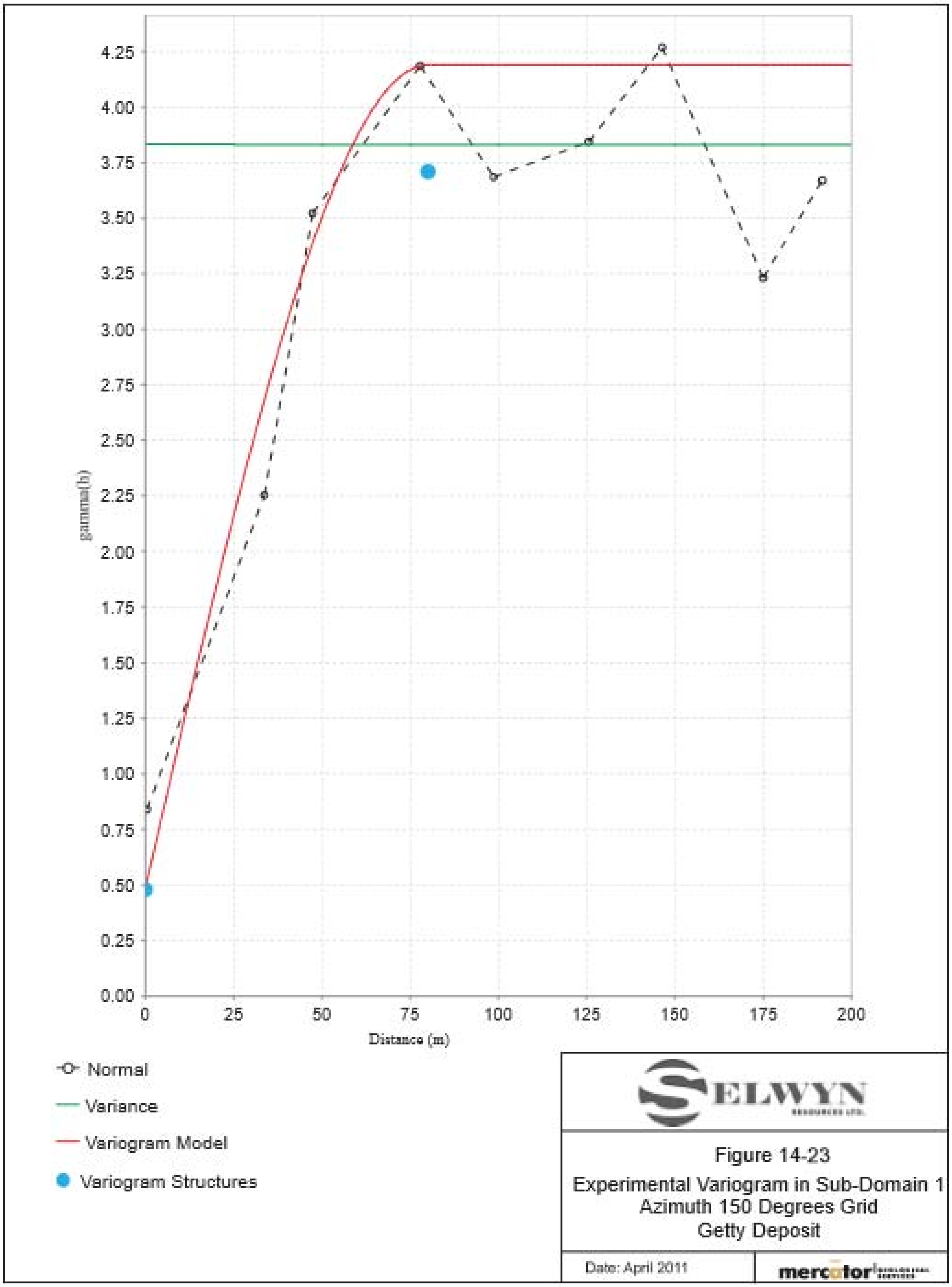
"Market conditions at the effective date of this report have changed since the 2008 resource estimate. Based on (1) review of London Metal Exchange 27 month forward contract pricing for lead and zinc, (2) consideration of current and future market pricing projections prepared for Selwyn (Brook Hunt, 2010), (3) availability of 2007-2008 milling recovery data from Scotia Mine, and (4) provision of relevant smelter return factors, the authors have chosen to redefine zinc equivalent for current purposes. Zinc Equivalent % (Zn Eq.%) for this report is defined as Zn % + (Pb % x 1.18), based on mill recoveries of 89.3% for zinc and 89.5% for lead, $US1.10/lb Zn and $US1.15/lb Pb metal pricing and smelter returns of 85% for Zn and 95% for Pb. A 2.00% Zn Eq. resource statement cutoff value was used and reflects open pit development potential.

26.17.3.5 Variography

"As reported by Cullen et al. (2008), an initial assessment of variography for the deposit area was carried out for historic drill hole data by creation of experimental variograms for combined zinc plus lead (zinc + lead) values for the largest northwest trending sub-domain of the deposit that corresponds with mineralization developed along the contact between overlying evaporite and extending southwest into the dolomitized bank proper. Further details pertaining to deposit sub-domains are presented in the following sections. In plan projection the selected sub-domain measures approximately 700 meters in length by 200 meters in average width and forms a broad corridor of northwest striking, flat-lying to northeast-dipping mineralized carbonate that shows restriction of most mineralization to a relatively narrow, 150 meter elevation interval. Local irregularities of the mineralized carbonate's trend are present in this corridor and take the form of promontories and re-entrants that have associated variations in strike and dip components.

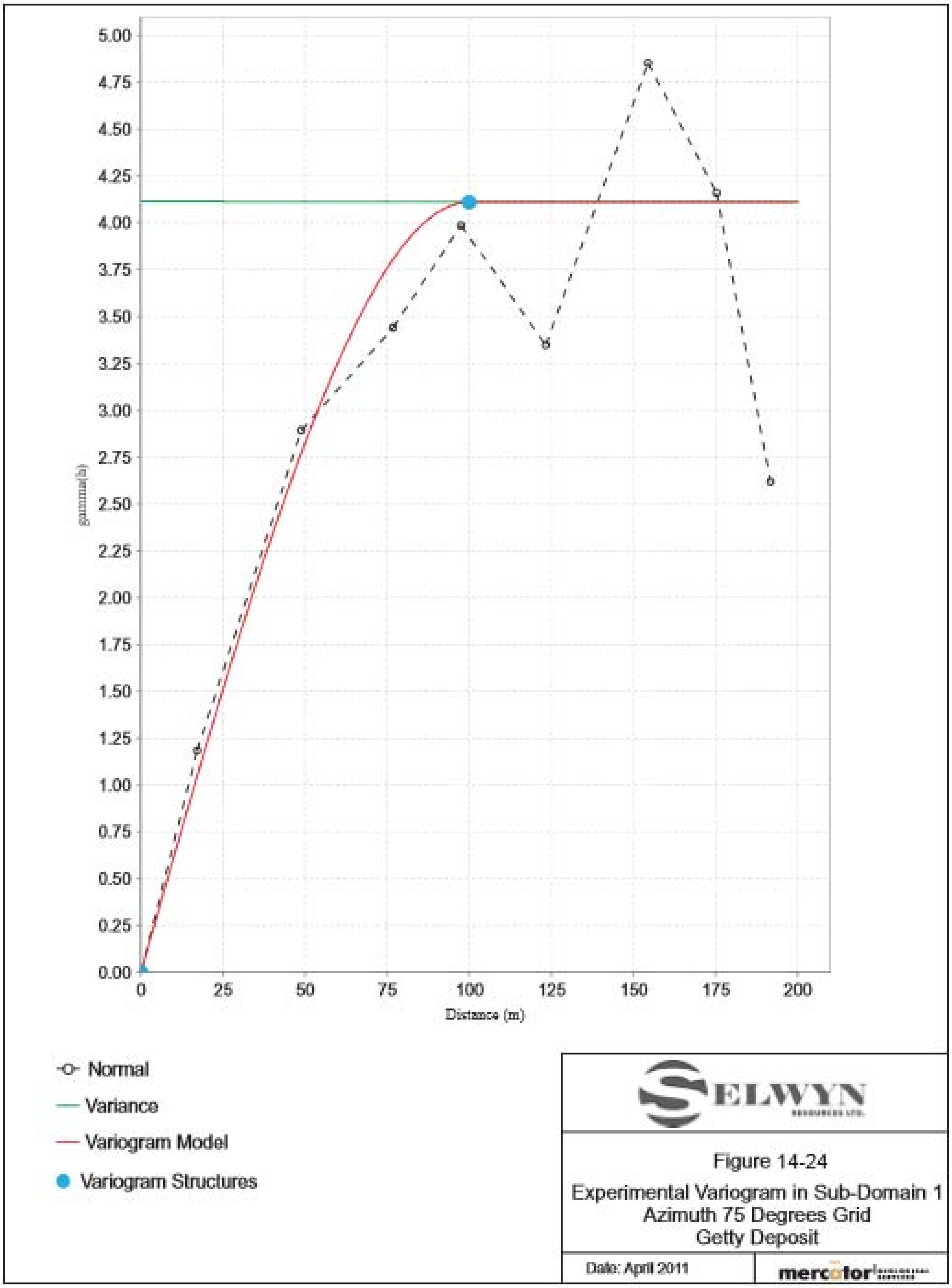
"Experimental variograms for the selected sub-domain were calculated at various lags and bearings within a horizontal reference plane and resulted in selection of spherical variogram models for major and semi-major axes of continuity in orientations that correspond to the dominant geological strike and dip directions within the sub-domain. Representative variogram models for these two axial components are presented in Figure 26-3 and Figure 26-44 and show ranges of 75 meters and 100 meters respectively. Experimental variograms were also calculated in the same horizontal reference plane for the entire composite data set occurring within the deposit peripheral constraint and these provided definition of spherical variogram models showing similar major and semi-major axis orientations as those calculated for the northwest sub-domain, but with higher degrees of complexity resulting from combination of data from the various orientation sub-domains present within the deposit.

Figure 26-3: Variogram Model Downhole



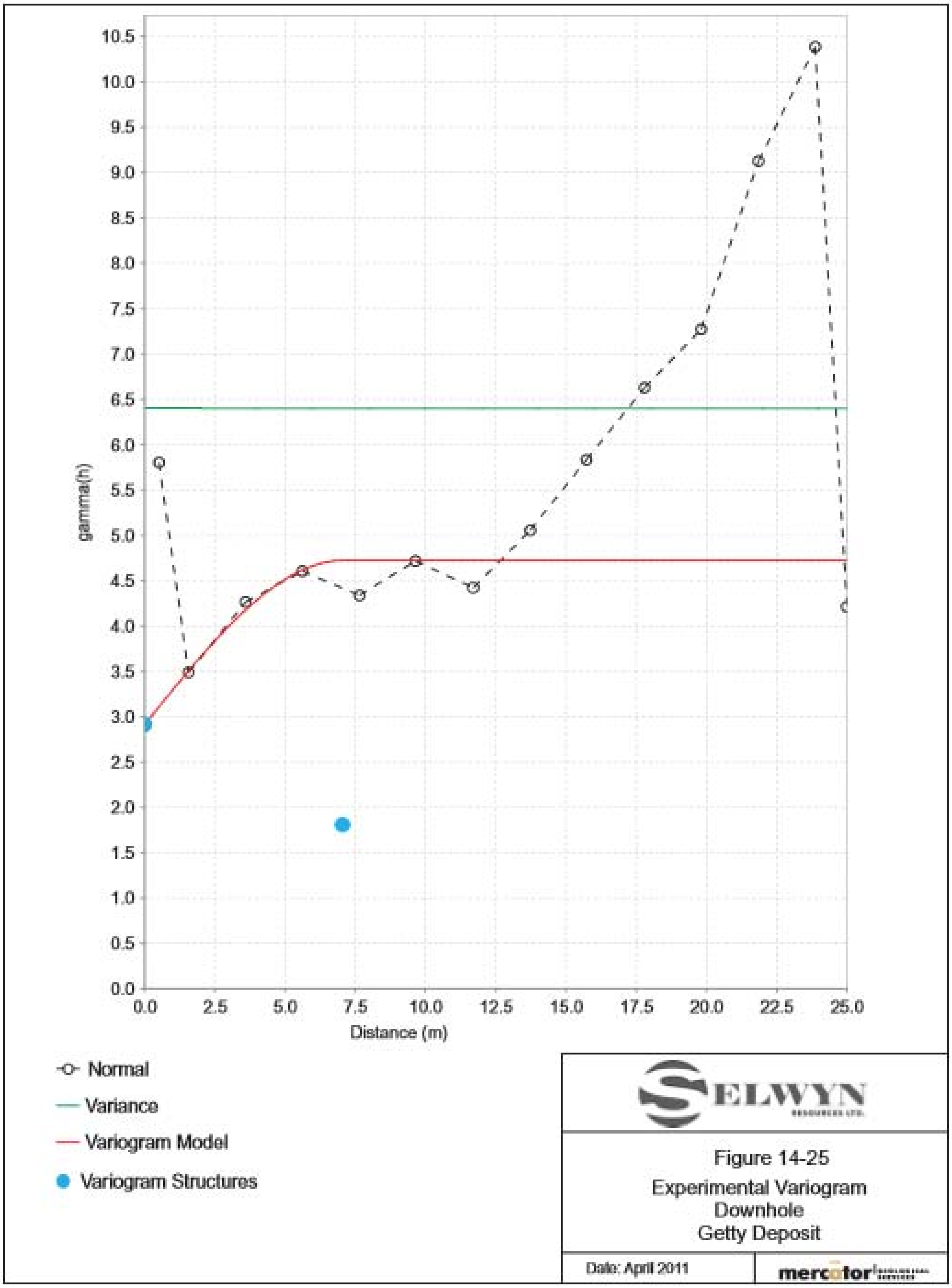
From Cullen et al, 2011.

Figure 26-4: Semi-Major Axis Variogram Model



From Cullen et al, 2011.

Figure 26-5: Grade Interpolation Downhole



From Cullen et al, 2011.

"Down hole experimental variograms and spherical model variograms were also prepared to assess grade continuity and correlation trends vertically within the dolostone unit that hosts the deposit. Figure 14-25 presents the best resulting down-hole variogram model and supports a range of 12 meters at a lag of 2 meters. This range is interpreted as reflecting the average mineralized thickness of the host carbonate within the deposit peripheral constraint and was considered during selection of a minor axis range value for the grade interpolation search ellipse.

"Ranges for variograms defined for the main northwest trending sub-domain were assumed to be applicable in the other deposit sub-domains, based on (1) correlation of the modeled continuity trends with local geological strike and dip directions and (2) independent confirmation of grade continuity based on systematic review and interpretation of multiple geological and assay cross sections through the deposit. In combination, these assumptions largely reflect the recognized stratabound character of the zinc and lead mineralization within the Gays River Formation host sequence in the Getty Deposit area.

26.17.3.6 Setup of 2011 Three Dimensional Block Model

"Block model total extents were defined in local grid coordinates as being from 6000 meters East to 7145 meters East and from 6300 meters North to 7150 meters North. The model extends in elevation from 150 meters to 700 meters relative to the Scotia Mine local grid that has a datum of mean sea level plus 500.11 meters. The nominal topographic surface in the Getty Deposit area occurs between the 550 meter and 520 meter local grid elevations and all resource solids respect the bedrock/overburden surface defined by the resource drill hole data set. As noted earlier, all drill holes in the Getty resource database are coordinated to both the Scotia Mine local grid and to UTM Zone 20 (NAD83) and collar coordinates for the local grid are reported in Appendix 2 of the (NI) 43-101 report "Updated Mineral Resource Report filed Oct 8 2012. The local grid closely reflects the 3° Modified Transverse Mercator (MTM) projection for Nova Scotia (ATS 77 datum).

"A standard block size for the model was established at 2.50 meters x 2.50 meters x 2.50 meters, with no sub-blocking. Descretization within blocks was 1 x 1x 1 and no block rotation was applied. The chosen block size reasonably reflects the character of mineralization within the deposit and also provides approximation of a mining unit size that could be applicable in development of this style of base metal deposit.

"All historic drill holes were lithocoded using the lithocode system originally established by Westminer for the Gays River Deposit. This system was also being used in the Getty Deposit drilling program by Acadian.

"Resource estimation was completely constrained within a peripheral deposit solid developed from wireframing of mineralized envelope limits on geological cross sections cut through the deposit. A minimum 1.0% (zinc + lead) value over a minimum 3.0 meter down hole sample length was used initially to define wire-framed mineralized envelope limits for a peripheral deposit constraint, with slight modifications made locally as required after inspection of the resultant solid. Lateral or down-dip deposit limits were typically created at midpoints between holes that mark the mineralized zone to non-mineralized zone transition or at a distance of 25 meters from a mineralized drill hole, the lesser distance being utilized.

"To properly accommodate deposit geometry during modelling, twenty-six grade interpolation sub-domains were established within the block model peripheral constraint and these are illustrated in (Figure 21). Sub-domains reflect areas of common geometric orientation of the mineralized carbonate and were established as discrete three dimensional constraints within which block grade interpolation could be carried out. Contributing composites for block grade interpolation were not constrained within the sub-domains, to ensure that modeling allowed grade continuity to exist across sub-domains boundaries. Fifteen sub-domains occur contiguously within the main northwest trending deposit outline and the remaining 11 occur contiguously within the southwest zone of the deposit that, at the minimum cut-off used in this report, has been modeled as a separate mineralized area immediately adjacent to the main deposit (refer to Figure 14-26).

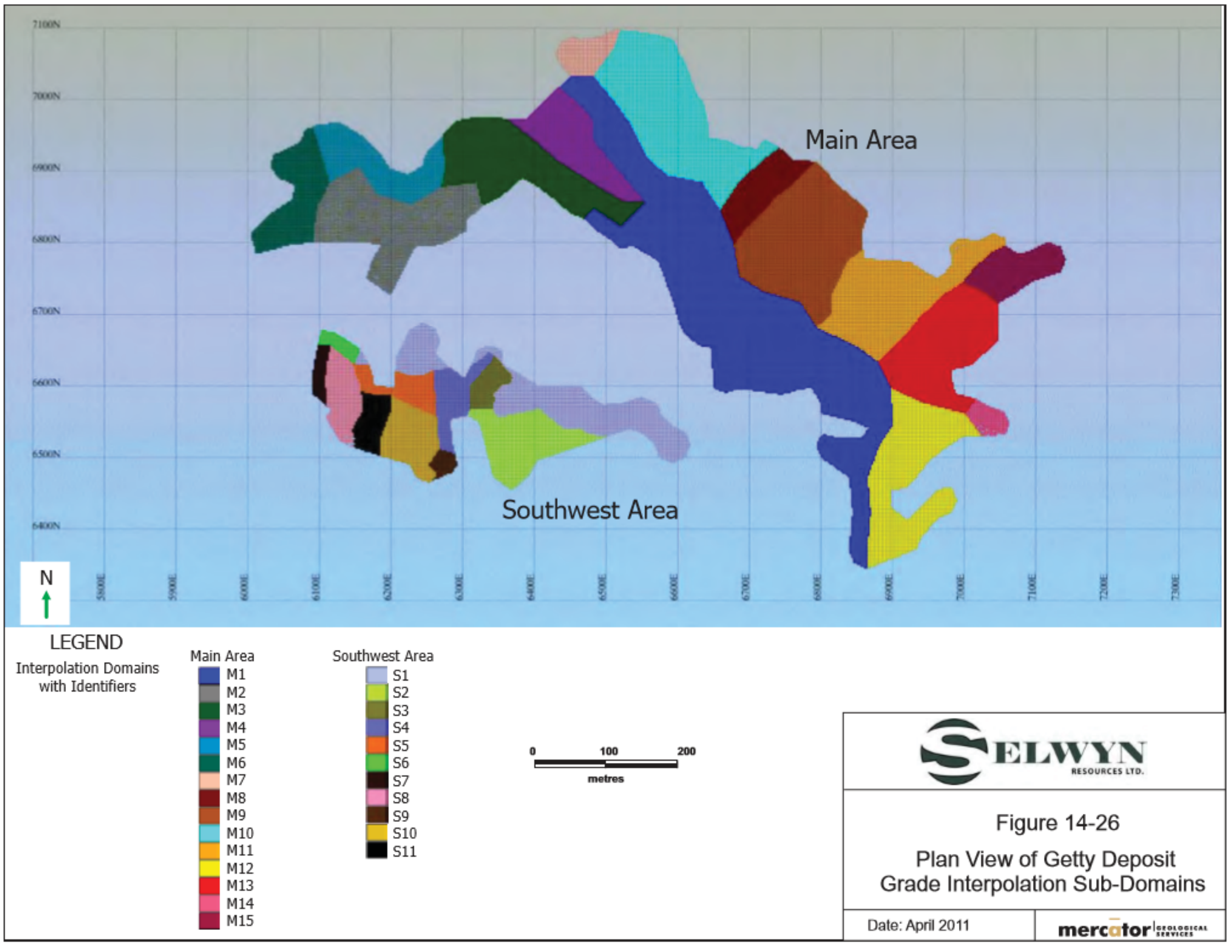
26.17.3.7 Assignment of Resource Estimate Cutoff Values

"A minimum cutoff value of 2.0 % zinc equivalent was used for reporting the current mineral resource estimate. This value was selected to reflect recognized potential for open pit development of the deposit and processing of ore at the adjacent Scotia Mine milling complex.

26.17.3.8 Material Densities

"No historic collection of Specific Gravity (SG) data for either the Scotia Mine or Getty Deposits was identified in historic records. However, during the course of the 2007-2008 drilling program, Mercator selected 120 dolostone and basal breccia pulp samples representing the grade range within the deposit and submitted these to ALS Chemex in Sudbury, ON for the purpose of Specific Gravity (SG) determination. Pyncometer and methanol laboratory methodology was utilized as set out in the ALS Chemex OA-GRA-08b laboratory protocol. Analytical results for zinc and lead had previously been received for all of the samples submitted for SG determination. No porosity factor was used in the specific gravity calculations.

Figure 26-6: Grade Interpolation Domains in Getty Deposit



From Cullen et al, 2011.

"Specific gravity (SG) values for the block model were assigned by calculation based on a base dolostone SG value of 2.82 g/cm3 and application of the formula set out below that assigns SG values based on zinc and lead block grades plus the base dolostone value. Zinc is assumed to be present as sphalerite and lead to be present as galena. This approach is consistent with methodology used for the previous Getty Deposit resource estimate by Mercator (Cullen et al., 2008) and follows the earlier example of MineTech International Limited (Roy et al., 2006) for calculation of mineral resources and reserves supporting the recent feasibility study for Acadian's adjacent Scotia Mine project.

"The 120 SG determinations from the Acadian drilling program were used to assess the assignment equation and results correlated sufficiently well to maintain its use. However, the equation was modified through increase of the original base dolostone SG value of 2.7 g/cm3 to 2.82g/cm3. SG values calculated for each block were multiplied by corresponding block volumes and results summed

according to applied cutoff parameters to obtain tonnage values for the deposit model. For purpose of review, descriptive statistics for calculated block density values used in the current deposit model are presented in Table 26-8.

Table 26-8: Descriptive Statistics: Block Model Density Values

Parameter	Value
Mean	2.86
Variance	0.001
Standard Deviation	0.028
Coefficient of Variation	0.010
Maximum	3.27
Minimum	2.82
Number	209757

26.17.3.9 Interpolation Ellipsoid and Resource Estimation

"Inverse Distance Squared (ID2) grade interpolation was used to assign block model metal grades, with blocks being fully constrained by limits of the 26 separate resource domain solids. Variogram models were used in conjunction with geological model attributes to guide assignment of major, semi-major and minor axis range values for interpolation ellipses used in the current model. Unique search ellipse orientation parameters were developed that reflect local geological strike and dip components for mineralized carbonate in each of the 26 interpolation domains and axial orientations were assigned to conform to this geometry.

"Major and semi-major axial range values for the ellipsoids were set at 75.00 meters for each domain and in no case exceeded the maximum major and semi-major range values indicated by the selected assay composite variogram models. The 75.00 meter range in both major and semi-major orientations was considered sufficient to insure block grade interpolation from 3 contributing drill holes in a 25 meter spaced drill pattern. Minor axis ranges of 37.5 meters were assigned to ensure full exposure to the thickness of stratabound mineralization within all sub-domains. This value exceeds the down-hole variogram range mentioned above and is fifty percent of the selected major and semi-major axis range values. Minor axis range selection was weighted on the basis of the deposit geological model to ensure inclusion of the full host sequence stratigraphic thickness in all sub-domains. Orientation parameters pertaining to the 26 grade interpolation sub-domains appear in Table 26-9 and Figure 14-27 presents a graphic representation of the various search ellipses superimposed on the block model.

Table 26-9: Search Ellipse Parameters for Interpolation Domains

Interpolation Domain Name	Azimuth (Degrees)	Plunge (Degrees)	Dip (Degrees)
Main 1	0	0	0
Main 2	0	0	0
Main 3	306	-22.5	-33.5
Main 4	306	-20.5	37
Main 5	0	-24	0
Main 6	250	-25	-18
Main 7	295	-33	0
Main 8	47	-31	35
Main 9	36	-20	-27
Main 10	33	-23	-10
Main 11	43	-15	30
Main 12	132	-24	15
Main 13	43	-8.5	-10
Main 14	0	0	0
Main 15	58	23	0
South 1	103	0	0
South 2	90	-5	-31.5
South 3	190	10	-20
South 4	176	26	16
South 5	108	0	-30
South 6	307	0	22
South 7	184	41	4
South 8	180	41	-45
South 9	193	44	7
South 10	194	38	-24
South 11	197	41	42

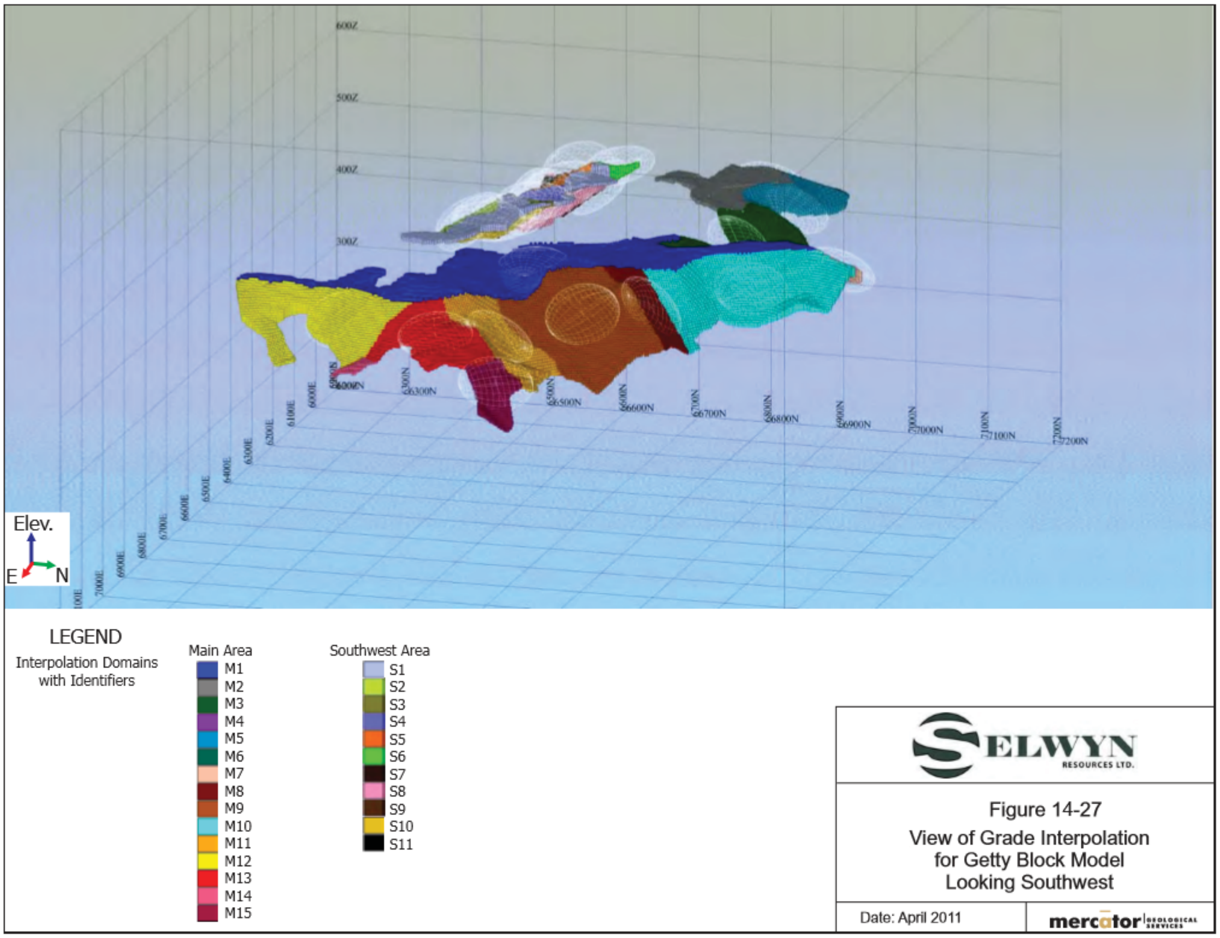
"A maximum of 12 included sample composites was established for estimation of individual block grades, with no more than 4 composites allowed from a single drill hole.

"These parameters ensured both multiple drill hole inclusion in block grade estimations and lateral grade projection between drill holes in dip and strike orientations. Single passes of ID2 grade interpolation were separately completed for the zinc and lead data sets within each of the 26 interpolation sub-domains and results were initially reported at grade cut-offs of 1.50%, 2.00%, 2.50% and 3.00% (zinc equivalent).

"Grade distribution within the block model was assessed against vertical geological and grade cross sections cut through the deposit at nominal spacings of 50 to 70 meters and also against horizontal sections cut through the model at 10 meter elevation intervals. Metal distribution trends observed in the sections were considered acceptable against the geological model Figure 26-7 though

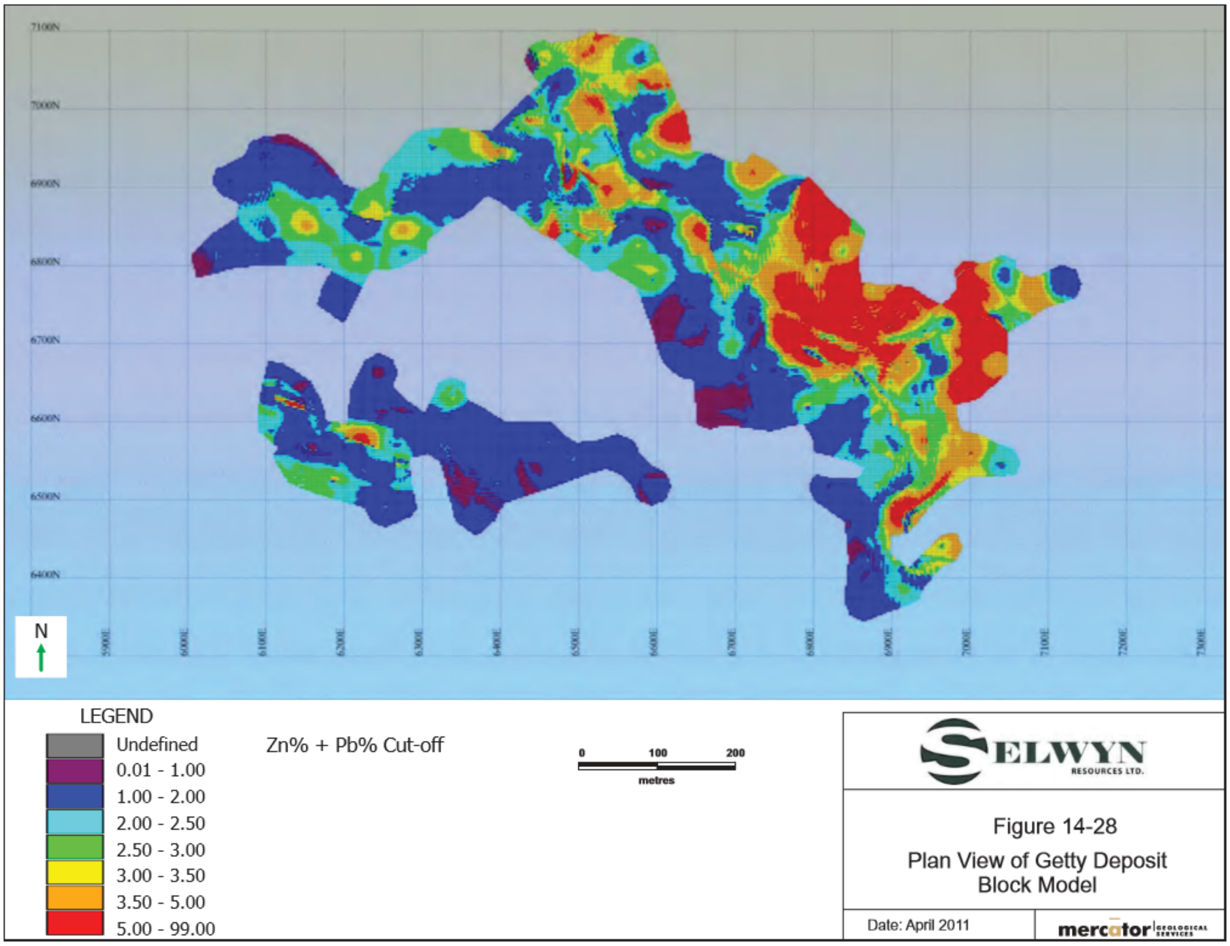
Figure 26-11 present perspective views of block model grade distribution trends at specified cut-off values.

Figure 26-7: View of Grade Distribution Looking Southwest



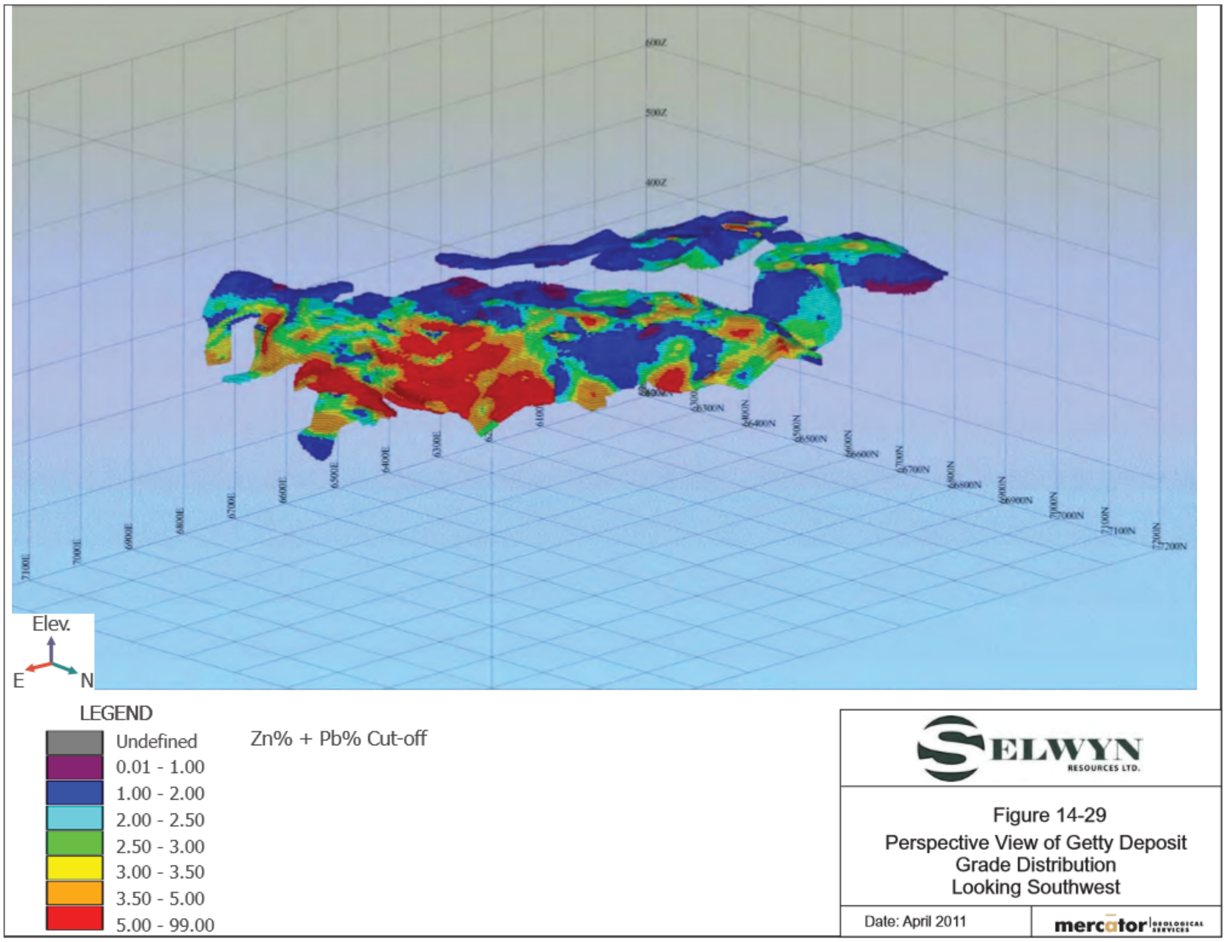
From Cullen et al, 2011.

Figure 26-8: Getty Deposit Grade Distribution Looking Northeast



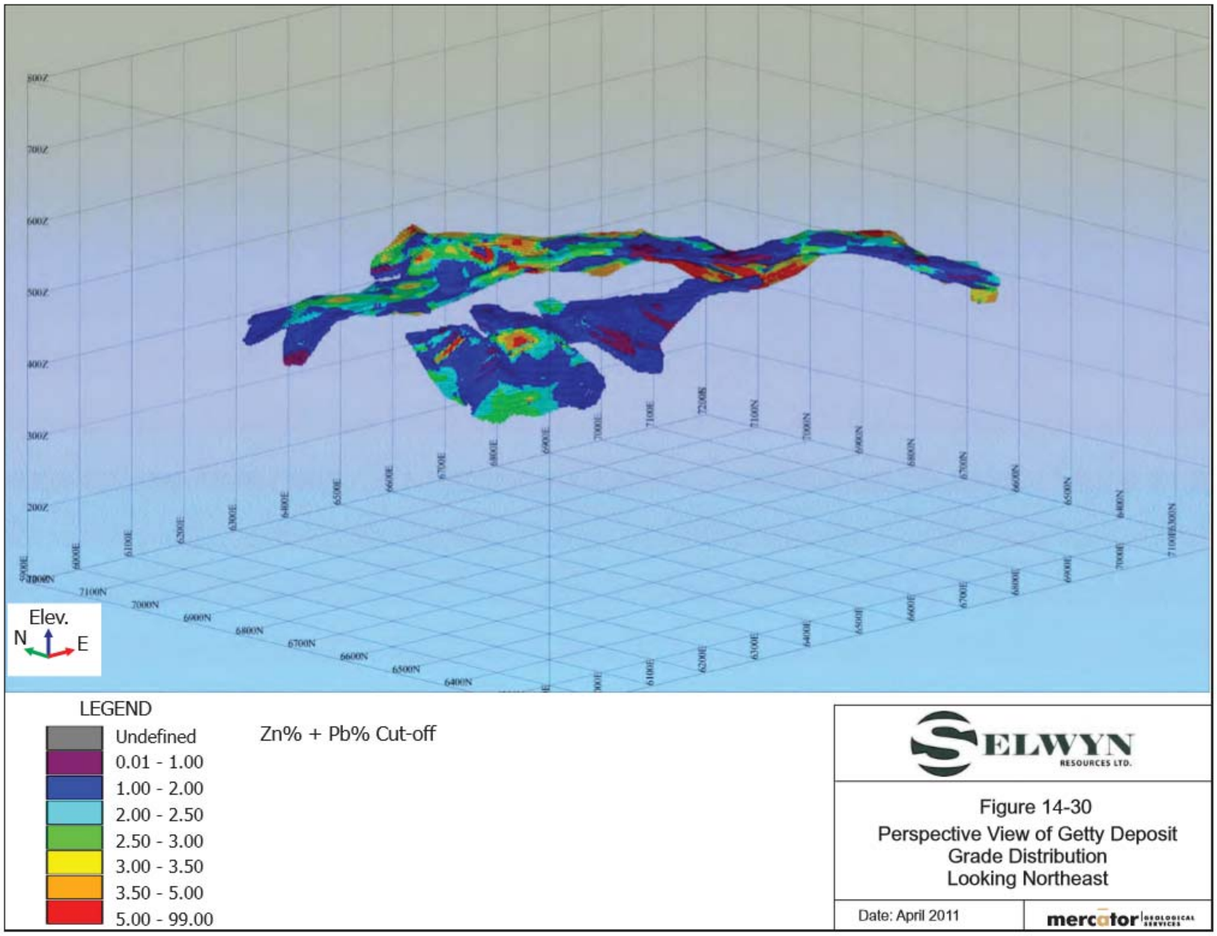
From Cullen et al, 2011.

Figure 26-9: Perspective View of Getty Deposit Grade Distribution Looking Northwest



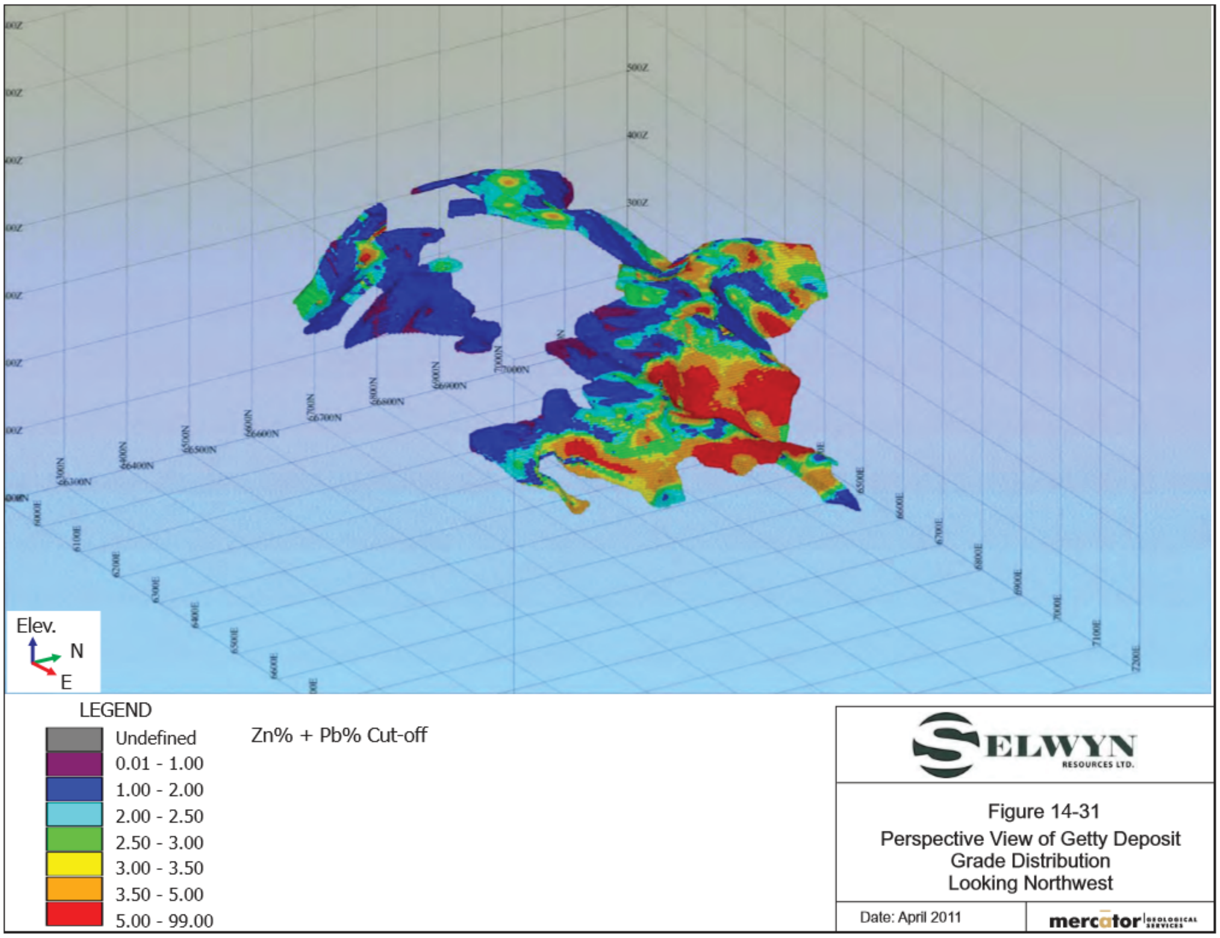
From Cullen et al, 2011.

Figure 26-10: Perspective View of Getty Deposit Grade Distribution Looking Northeast



From Cullen et al, 2011.

Figure 26-11: Perspective View of Getty Deposit Grade Distribution Looking Northwest



From Cullen et al, 2011.

26.17.3.10 Resource Classification

"Mineral resources presented in the current estimate have been assigned Inferred, Indicated and Measured resource categories that reflect increasing levels of confidence with respect to spatial configuration or resources and corresponding grade assignment within the deposit. Several factors were considered in defining resource category assignments, including drill hole spacing, geological interpretations and integrity of supporting data sets. Results of the 2007-2008 core drilling program by Acadian provided the most important upgrading factor to the deposit data set in comparison to the 2007 resource estimate which previously reported Inferred mineral resource. The new Acadian drill holes provided a nominal drill hole spacing of approximately 50 meters by 50 meters over much of the deposit area and constituted a major degree of infilling with respect to more broadly spaced historic drill holes that supported the previous estimate. The increased drill hole density factor was augmented by

additional QA/QC program results associated with twinning of 10 historic drill holes during the 2007-2008 Acadian drill program and also by re-logging and sampling of 10 historic drill holes for which archived core was available. Positive results from all noted programs served to upgrade overall confidence in the project data set and justified definition of higher category resources.

"Definition parameters for each resource category specified in the current Getty estimate are set out below and Figure 14-32 illustrates distribution of categories in plan view.

"Measured Resources: All blocks with grades based on three drill holes and a minimum of 9 included samples, with not more than 4 composites from a single drill hole, for which the averaged distance to included samples was 28 meters or less with no sample greater than 50% of the major axis range (37.5m) from the block were categorized as Measured mineral resources.

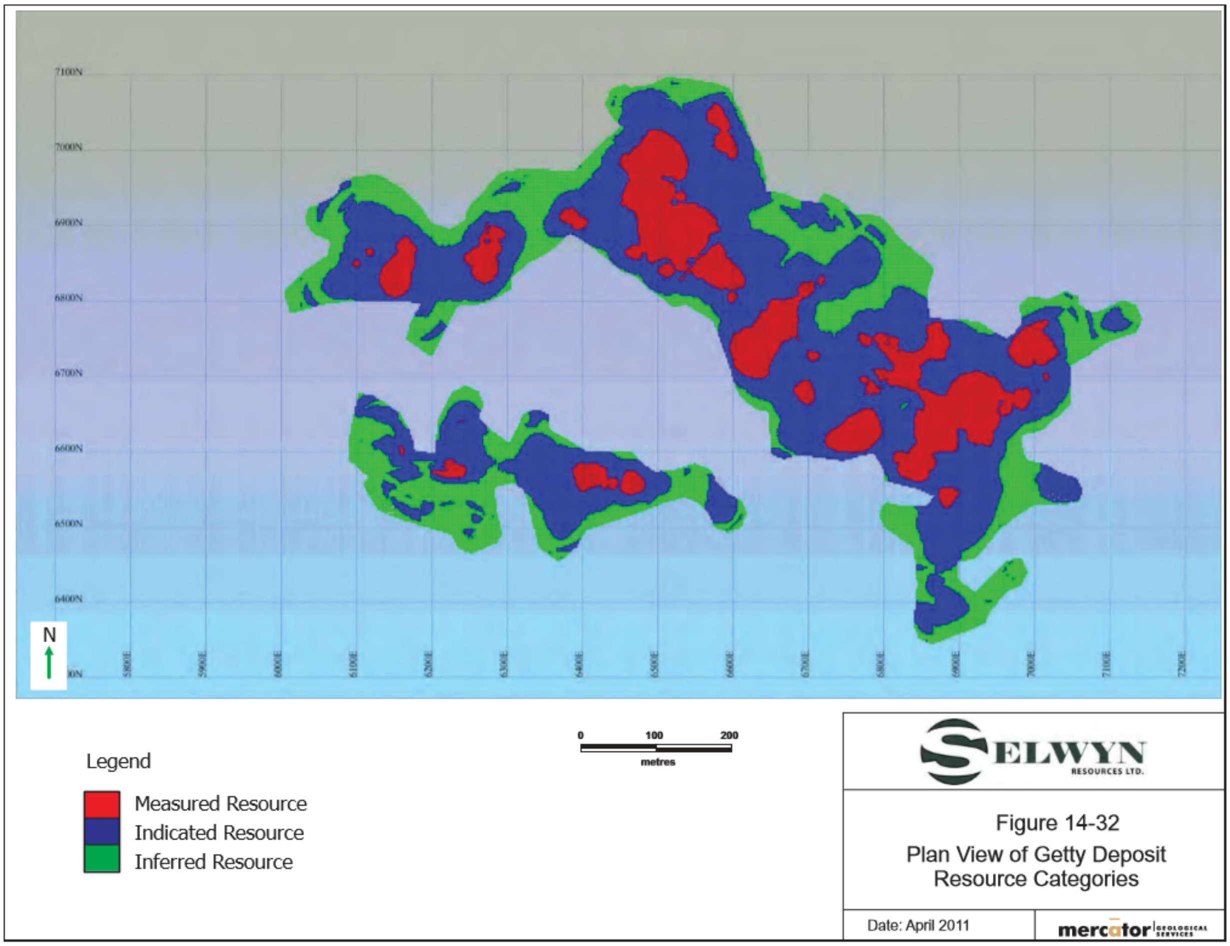
"Indicated Resources: All blocks with grades based on two or more drill holes and a minimum of 5 included samples, with not more than 4 composites from a single drill hole, for which the averaged distance to included samples was 40 meters or less with no sample greater than 75% of the major axis range (56.5m) from the block were categorized as Indicated mineral resources.

"Inferred Resources: All blocks present within the deposit solid that did not meet other resource category requirements and for which interpolated grades were present were categorized as Inferred mineral resources.

26.17.3.11 Statement of Mineral Resource Estimate at Effective Date

Table 26-10 presents a statement of the updated mineral resource estimate for the Getty Zn-Pb deposit supported by content of this technical report. The estimate is considered to be compliant with both the CIM Standards and disclosure requirements of NI 43-101. The effective date of the estimate is deemed to be 30 March 2011. All parameters utilized in the 2008 resource estimate were applied to this revised estimate with the exception of the Zn Equivalent % calculation factor. For the current resource estimate Zn Equivalent % (Zn Eq %) has been defined as Zn % + (Pb % × 1.18) and is based on mill recoveries of 89.3% for Zn and 89.5% for Pb, $US1.10/lb Zn and $US1.15/lb Pb metal pricing and smelter returns of 85% for Zn and 95% for Pb.

Figure 26-12: Plan View of Getty Deposits Resource Categories



From Cullen et al, 2011.

Table 26-105: Mineral Resource Estimate for Getty Deposit- 30 March 2011

Getty Deposit - Resource Statement - Zn Eq. % * Cut-off					
Resource Category	**Zn Eq. % Cut-off**	**Tonnes (Rounded)**	**Zn %**	**Pb %**	**Zn Eq %***
Measured	1.50	1,930,000	1.81	1.26	3.30
Indicated	1.50	3,790,000	1.62	1.21	3.05
Indicated + Measured	**1.50**	**5,720,000**	**1.68**	**1.23**	**3.13**
Inferred	**1.50**	**1,350,000**	**1.52**	**1.31**	**3.06**
Measured	***2.00**	**1,550,000**	**1.97**	**1.45**	**3.68**
Indicated	***2.00**	**2,810,000**	**1.82**	**1.44**	**3.51**
Indicated + Measured	****2.00***	***4,360,000***	***1.87***	***1.44***	***3.57***
Inferred	****2.00***	***960,000***	***1.73***	***1.59***	***3.60***
Measured	2.50	1,180,000	2.14	1.68	4.12

Getty Deposit - Resource Statement - Zn Eq. % * Cut-off					
Resource Category	**Zn Eq. % Cut-off**	**Tonnes (Rounded)**	**Zn %**	**Pb %**	**Zn Eq %***
Indicated	2.50	1,950,000	2.06	1.70	4.07
Indicated + Measured	***2.50***	***3,130,000***	***2.09***	***1.69***	***4.09***
Inferred	***2.50***	***680,000***	***1.95***	***1.88***	***4.16***
Measured	3.00	860,000	2.34	1.95	4.64
Indicated	3.00	1,300,000	2.35	2.03	4.74
Indicated + Measured	***2.50***	***2,160,000***	***2.35***	***2.00***	***4.70***
Inferred	***3.00***	***460,000***	***2.21***	***2.23***	***4.85***

Notes: (1) Zn Equivalent % (Zn Eq.%) = Zn % + (Pb % × 1.18) and is based on mill recoveries of 89.3% for Zn and 89.5% for Pb, $US1.10/lb Zn and $US1.15/lb Pb metal pricing and smelter returns of 85% for Zn and 95% for Pb, (2)* denotes the 2.00% Zn Eq. resource statement cutoff value that reflects open pit development potential

26.17.3.12 Validation of Model

Comparison to Geological Sections

"Results of block modeling were compared on a section by section basis with corresponding interpreted geological and grade distribution sections prepared prior to block model development. This showed that block model grade patterns show good correlation with those interpreted from the geological sections and that the stratabound character of the mineralization was being properly represented. Results of visual inspection are interpreted as showing an acceptable degree of consistency between the block model and the independently derived sectional interpretation, thusly providing a measure of validation against the geological model developed for the deposit.

Comparison of Composite Database and Block Model Grades

"Descriptive statistics were calculated for those portions of the drill hole composite population falling within the total deposit peripheral constraint and these figures were compared to corresponding values calculated for the resource estimate block model. Results of the comparison are tabulated in Table 14-16. Mean drill hole assay composite grades for zinc and lead compare closely with corresponding zinc and lead grades calculated for the entire block model and provide a check on bias within the model with respect to the underlying total assay composite population.

Table 26-11: Comparison of Drill Hole Assay Composite and Block Model Grades

Parameter	*Total Model Grade (Zn%)	*Total Model Grade (Pb%)	Composites Grade (Zn%)	Composites Grade (Pb%)
Mean	1.43	1.01	1.46	1.00
Variance	0.86	0.99	1.94	2.53
Standard Deviation	0.93	1.00	1.39	1.59
Coef. of Variation	0.648	0.990	0.948	1.580
Maximum	10.27	14.52	11.30	18.54
Minimum	0.00	0.00	0.00	0.00
Number	209,757	209,757	1961	1961

*Defined as all blocks having interpolated grades within the deposit peripheral constraint

Comparison of With Nearest Neighbour Grade Interpolation Model

"The ID2 block model was checked using Nearest Neighbour (NN) grade interpolation methodology within the same resource solids used for the ID2 method and associated weighted average drill hole intercepts appear in Appendix 2 of NI 43-101 Updated Mineral Resource Report, filed October 8 2012. Assigned block resource categories were constant between models as were metal cut-off values. Results of the NN estimation appear in Table 26-12 and Figure 14-3 provides a comparison to ID2 model results.

Table 26-12: Results of Nearest Neighbour Block Model Estimate

Cutoff: Pb% + Zn%	Resource Category	Tonnes (Rounded)	Pb %	Zn%	Pb%+Zn%
2.00	Measured	1,480,000	1.44	1.90	3.34
2.00	Indicated	2,320,000	1.51	1.96	3.47
2.00	*Indicated Plus Measured*	*3,800,000*	*1.48*	*1.94*	*3.42*
2.00	Inferred	880,000	1.58	1.81	3.39
2.50	Measured	1,050,000	1.75	2.07	3.82
2.50	Indicated	1,490,000	1.90	2.27	4.17
2.50	*Indicated Plus Measured*	*2,540,000*	*1.84*	*2.19*	*4.03*
2.50	Inferred	530,000	2.05	2.15	4.20
3.00	Measured	700,000	2.07	2.31	4.38
3.00	Indicated	1,080,000	2.19	2.56	4.75
3.00	Indicated Plus Measured	1,780,000	2.14	2.46	4.60
3.00	Inferred	410,000	2.24	2.41	4.64

Table 26-13: Comparison of Block Model Interpolation Methods

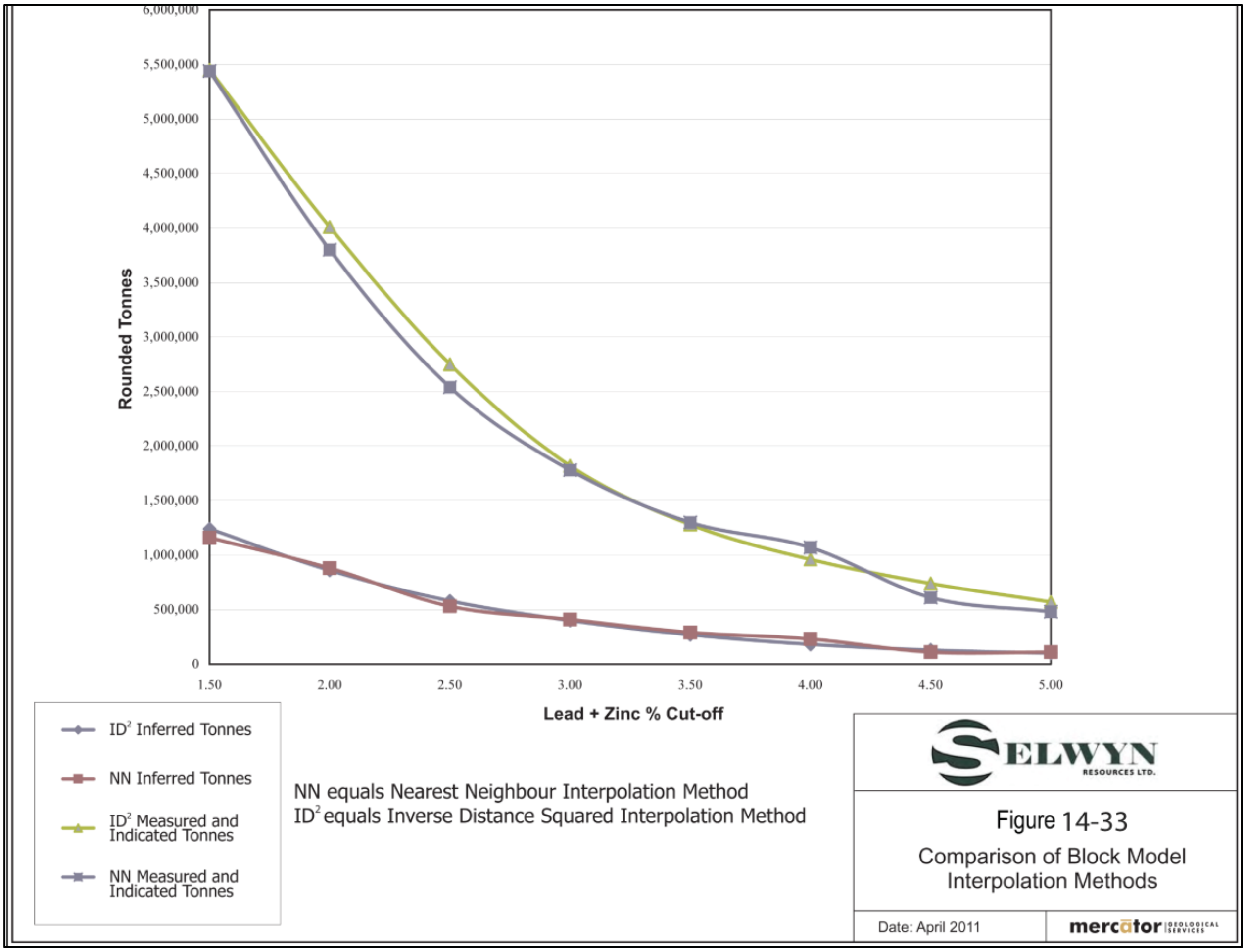


Figure 14-33
Comparison of Block Model Interpolation Methods

"Grade and tonnage figures for the two block models correlate well at all cutoff values and are interpreted as providing an acceptable check of the ID2 model.

26.17.4 Comments on Previous Resource or Reserve Estimates

"Three historic mineral resource estimates were reviewed for purposes of this report and these were referenced previously in section 5.2. The first was prepared in 1976 for Getty by MPH Consulting Limited (Riddell, 1976) and apparently followed earlier in-house estimates by Getty. Subsequently, an in-house assessment was prepared by Esso (MacLeod, 1980) and in 1992 Westminer also completed an estimate (Hudgins and Lamb, 1992). Results of these programs are presented in Table 14-18 and, as noted earlier, all are historic in nature, pre-date NI 43-101 and are not compliant with current CIMM Standards. As such, they should not be relied upon.

Table 26-14: Historic Tonnage and Grade Estimates for Getty Deposit

(Estimates Are Not Compliant with NI 43-101 or CIM Standards)

Reference	Cutoff	Tonnes	Pb %	Zn %	Zn + Pb %
Riddell (1976)	2% Zn + Pb	4,005,000	1.84	1.87	3.02
*MacLeod (1980)	1.5% Zn +Pb	3,078,000	1.37	1.60	2.97
**Hudgins and Lamb (1992)	**1.5% Zn Eq.	4,500,000	1.33	1.87	3.20

* Diluted and Minable; **Zn Eq. = Zn% + 0.60 x Pb%

Notes: With regard to the historic mineral resource estimates stated above 1) a qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves; 2) the issuer is not treating the historical estimate as current mineral resources or mineral reserves as defined in Sections 1.2 and 1.3 of NI 43-101; and 3) the historical estimate should not be relied upon.

"Support documents provided for the historic estimates showed that those of Getty and Esso were based on drill-hole-centered polygonal methods with tonnage weighting to establish final deposit grade. A single density factor of 11.5 cubic feet per ton (~2.78g/cm3) was used in the Riddell (1976) estimate and this appears to have been used by MacLeod (1980) before application of a 10% tonnage reduction factor to drill hole intercepts. Westminer employed a cross-sectional method using Surpac® mining software to determine resource area limits and volume and used a single density factor of 2.75 g/cm3 to estimate tonnage. Deposit grade was calculated as the length-weighted average of all drill hole intercepts, but spatial distribution of grade within the deposit was not specifically addressed.

"A summary review of supporting file information for the historic estimates was completed for current purposes and it is apparent that the noticeably lower tonnage figure quoted by Esso reflects exclusion of certain drill holes based on the report's development potential assumptions. The higher lead grade in the MPH estimate is also notable but main contributing factors were not clearly identified.

"Riddell (1976) completed a preliminary economic assessment for open pit development of a 3.6 million ton (3.3 million tonne) portion of the deposit at a diluted grade of 1.28% Pb and 1.74% Zn. Modeling parameters included options of a stand-alone mill, custom milling of ore at Esso's adjacent Gays River site and development of a jointly-owned mill complex in association with Esso. Analysis showed that a 20 year model producing at 182,000 tons per year with a dedicated mill was uneconomic. However, 10 year projects producing at 375,000

tons per year were financially attractive in both the custom milling and jointly owned mill models.

"In 1980 Esso reported on economic aspects of developing the deposit based on an insitu tonnage and grade model of 3.1 million diluted tons (2.8 million tonnes) grading 1.37% Pb and 1.60% Zn (MacLeod, 1980). This study concluded that mining the deposit through open-pit methods as an ore supplement to the Gays River deposit was economically viable, provided that important operating assumptions were met. Positive Net Present Value figures at 15% discounting were returned for 1000 and 1250 ton per day production rates, with the Gays River operation absorbing certain operating and capital cost components. George (1985) again reviewed deposit economics for Getty and used economic analysis applied to tonnage and grade curves to show that a deposit size of approximately 8 million tons was necessary to justify stand-alone profitable development at realizable metal grades. The earlier MPH work was also reviewed and some of the economic models updated. None of the work indicated that profitable stand-alone development of the deposit could be expected under existing market conditions of the time.

"Hudgins and Lamb (1992) reported on preliminary economic analysis of a 3.9 million tonne portion of the total resource at their assigned grade and concluded that a positive economic case could be made for development of the property as a "top-up" source of feed for the Gays River concentrator. Assumptions included sharing of various operating costs with the Gays River operation and that the full 1500 tonne per day capacity of the Gays River concentrator would not be required for underground production.

"In review, each of the historic estimates reflects specific assumptions considered appropriate at the time of preparation. This includes exclusion of certain historic drill holes, establishment of different maximum depth criteria and use of differing minimum grade and width cut off values. The current estimate does not directly reflect any of the parameter sets used in the early programs and results are therefore different. However, all historic programs model the Getty Deposit as a relatively low grade accumulation of lead and zinc having potential for open pit development. From the grade and tonnage perspective the earlier estimates are generally consistent with results of the current estimate and provide relevant views of the deposit under historic market conditions.

"The first NI 43-101 compliant Getty resource estimate completed by Mercator for Acadian (Cullen et al. 2007) was based solely on historical drilling and the entire resource was assigned to the Inferred resource category. Inferred designation reflected drill hole spacing and historical nature of the supporting database. The

associated block model provided a well developed view of geological and grade trends within the deposit area and also highlighted the need to carry out a substantial amount of infill drilling before higher category resources could be defined for the deposit. Table 26-15 presents results of the Cullen et al. (2007) resource estimate, which, on a total tonnage basis, is approximately 19% smaller than total tonnage at the same cutoff value for the 2008 resource at comparable average grades.

Table 26-15: Getty Deposit Mineral Resource Estimate - December 2007*

Resource Category	Zn Equivalent % Threshold**	Tonnes (Rounded)	Lead %	Zinc %	Zinc% + Lead %
Inferred	2.00	4,160,000	1.40%	1.81%	3.21%
Inferred	2.50	2,860,000	1.60%	2.06%	3.66%
Inferred	3.00	1,970,000	1.82%	2.26%	4.08%
Inferred	3.50	1,300,000	2.09%	2.42%	4.51%

Notes:* Estimate is compliant with NI 43-101 and CIM Standards; ** Zn Equivalent calculated as Zn Equivalent = (Zn% + Pb %)

"Completion of infill drilling was recommended and ultimately carried out during the 2007-2008 Acadian drilling campaign that totalled 138 holes in the deposit area. Addition of results for the 138 drill holes is the principal difference between the 2008 resource data set and that used in the 2007 estimate, with the designation of higher category resources in reflecting increased confidence in deposit geology and grade distribution models (Cullen et al., 2008). The NI 43-101 compliant 2008 estimate is summarized in Table 26-6.

Table 26-16: Getty Deposit Mineral Resource Estimate - November 2008*

Resource Category	Zinc% +Lead% Threshold**	Tonnes (Rounded)	Lead %	Zinc %	Zinc% + Lead %
Measured	2.00	1,470,000	1.48	2.02	3.50
Indicated	2.00	2,540,000	1.48	1.91	3.39
Indicated Plus Measured	***2.00***	***4,010,000***	***1.48***	***1.95***	***3.43***
Inferred	2.00	860,000	1.65	1.82	3.48
Measured	2.50	1,070,000	1.74	2.22	3.97
Indicated	2.50	1,680,000	1.78	2.21	3.99
Indicated Plus Measured	***2.50***	***2,750,000***	***1.76***	***2.21***	***3.98***
Inferred	2.50	580,000	1.98	2.09	4.07
Measured	3.00	740,000	2.04	2.47	4.52
Indicated	3.00	1,080,000	2.13	2.54	4.67
Indicated Plus Measured	***3.00***	***1,820,000***	***2.09***	***2.51***	***4.61***
Inferred	3.00	400,000	2.34	2.37	4.71

* Estimate is compliant with NI 43-101 and CIM Standards; ** Zn Equivalent calculated as Zn Equivalent = (Zn% + Pb %)

"A portion of this tonnage increase is directly attributable to change in base SG value for the block model, from 2.7 g/cm3 in 2007 to 2.82 g/cm3 in 2008. The remaining change is attributed to incremental extension of local deposit limits on the basis of 2007-2008 drilling program results."

26.18 Summary of Mineral Resources – Getty Deposit

The Gays River Deposit's mineral resource estimate was prepared by Doug Roy, M.A.Sc., P. Eng., and Tim Carew, P. Geo. of MineTech International Limited. The Getty Deposit's mineral resource estimate was prepared by Cullen et al. (2011) of Mercator Geological Services. The estimates were separately prepared using slightly different parameters, the most significant of which were different Zn equivalent grade formulae and different block cut-off grades for resource reporting.

26.18.1 Getty Deposit

Using a Zn-equivalency ratio of 1% Pb = 1.17% Zn and a block cutoff grade of 2% Zn-equivalent, Cullen et al. (2011) determined that Measured plus Indicated mineral resources totaled 4.4 million tonnes with average grades of 1.9% Zn and 1.4% Pb (refer to Table 26-17). Inferred mineral resources totaled 1.0 million tonnes with average grades of 1.7% Zn and 1.6% Pb.

Table 26-17: Getty Deposit Mineral Resources (from Cullen et al, 2011)

Resource Category	Zn Eq. % Cut-off	Tonnes (Rounded)	Zinc %	Lead %	Zinc Eq %*
Measured	2.00	1,550,000	1.97	1.45	3.68
Indicated	2.00	2,810,000	1.82	1.44	3.51
Indicated + Measured	2.00	4,360,000	1.87	1.44	3.57
Inferred	2.00	960,000	1.73	1.59	3.60

Notes: (1) Zn Equivalent % (Zn Eq.%) = Zn % + (Pb % x 1.18) and is based on mill recoveries of 89.3% for Zn and 89.5% for Pb, $US1.10/lb Zn and $US1.15/lb Pb metal pricing and smelter returns of 85% for Zn and 95% for Pb.

26.18.2 Gays River and Getty Deposits Combined

A summary of the mineral resources for both deposits was prepared. The reader is warned that the Gays River and Getty mineral resource estimates were prepared by different authors using different parameters.

Table 26-18: Combined Mineral Resources, Gays River and Getty Deposits*

Resource Category	Zn Eq. % Cut-off	Tonnes (Rounded)	Zinc %	Lead %	Zinc Eq %*
Measured	2.00	1,550,000	1.97	1.45	3.68
Indicated	2.00	2,810,000	1.82	1.44	3.51
Indicated + Measured	*2.00*	*4,360,000*	*1.87*	*1.44*	*3.57*
Inferred	*2.00*	*960,000*	*1.73*	*1.59*	*3.60*

Notes: (1) Zn Equivalent % (Zn Eq.%) = Zn % + (Pb % x 1.18) and is based on mill recoveries of 89.3% for Zn and 89.5% for Pb, $US1.10/lb Zn and $US1.15/lb Pb metal pricing and smelter returns of 85% for Zn and 95% for Pb.

26.18.3 Gays River and Getty Deposits Combined

A summary of the mineral resources for both deposits was prepared. The reader is warned that the Gays River and Getty mineral resource estimates were prepared by different authors using different parameters.

Table 26-19: Combined Mineral Resources, Gays River and Getty Deposits

Resource Category	Zn Eq. % Cut-off	Tonnes (Rounded)	Zinc %	Lead %	Zinc Eq %*
Measured	Varies	3,625,000	2.64	1.58	4.54
Indicated	Varies	8,580,000	2.82	1.61	4.75
Measured+Indicated	**Varies**	**12,205,000**	**2.76**	**1.60**	**4.68**
Inferred	Varies	4,637,000	2.22	1.53	4.05

* 1% Lead = 1.2 % Zinc.

26.19 Mining Methods

Mining will be done using conventional truck and shovel mining methods in the open pits, and Cut-and-Fill mining in the underground.

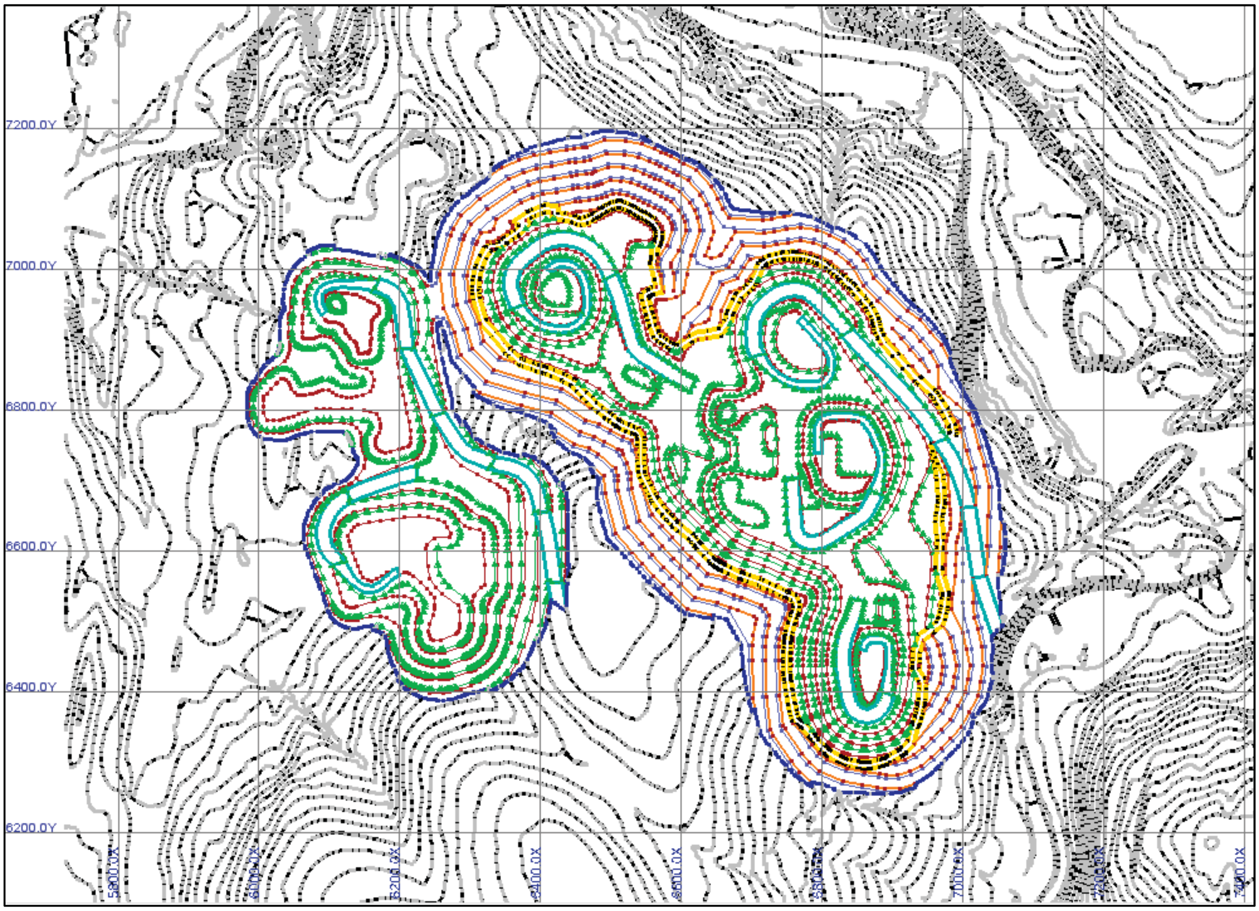
The open pits will mostly be mined on 10 m-high benches using 2 Cat 6018, 10 m^3, hydraulic shovels and Cat 777, 90 t-capacity haul trucks. A smaller Cat 390D, 4.6 m^3, hydraulic excavator will be used to mine and separate the resource. The use of the smaller excavator for resource mining should help reduce dilution, increase mine recovery, and improve ore segregation, for blending purposes. Drilling and blasting will be required for the rock portion of the deposit while the overlying overburden, which makes up approximately 60% of the waste to be mined, is considered free digging and will not require blasting.

The underground operation will access from the lower benches of the open pits to reduce waste development costs and to use the open pit excavations and facilities for water management.

26.19.1 Open Pit Design

A pit optimization and pit design analysis was performed in 2011 as part of the Gays River and Getty Deposit evaluation (see Figure 26-13). The pit design was not included in the 2011 PEA. It is shown in this report as an opportunity.

Figure 26-13: Getty Pit Design



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28.0 DATE AND SIGNATURE PAGE

Dated at Fall River, NS Effective Date: 02 February 2018 Date Signed:	***[Original signed and sealed by]*** ***[Jason Baker]*** Jason Baker, P. Eng. Author and Consultant, Mining
Dated at Fredericton, NB Effective Date: 02 February 2018 Date Signed:	***[Original signed and sealed by]*** ***[Jennifer L. McPhail]*** Jennifer L. McPhail, M. Eng., P. Eng. Author and Associate, Environmental
Dated at Sudbury, ON Effective Date: 02 February 2018 Date Signed:	***[Original signed and sealed by]*** ***[Kim Trapani]*** Kim Trapani, MBA, P. Eng. Author and Intermediate Engineer
Dated at Sudbury, ON Effective Date: 02 February 2018 Date Signed:	***[Original signed and sealed by]*** ***[Michael Romaniuk]*** Michael Romaniuk, P. Eng. Author and Senior Associate
Dated at Sudbury, ON Effective Date: 02 February 2018 Date Signed:	***[Original signed and sealed by]*** ***[Stephen J. Oak]*** Stephen J. Oaks, P. Eng. Author and Project Specialist

29.0 CERTIFICATES OF AUTHORS AND QUALIFIED PERSONS

CERTIFICATE OF AUTHOR

To: ScoZinc Mining Ltd. (“**ScoZinc**”)

I, Jason Baker, P.Eng., of Fall River, NS, do hereby certify that:

1. I am a Consultant employed with Stantec Consulting Ltd. (“**Stantec**”), working out of the Dartmouth, NS office.
2. This certificate applies to the Technical Report entitled “ScoZinc Mine – Preliminary Economic Assessment Update – Document No. RPT-17698” dated 02 February 2018.
3. I am a graduate of the University of Dalhousie, (B.Eng, 2000). I am a member in good standing with Engineers Nova Scotia, licence number 9627. My relevant experience is approximately 12 years of operational experience within the mining industry, as well as approximately 8 years of consulting to the mining industry on a wide range of projects. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
4. I have inspected the Property.
5. I am responsible for Sections 7, 8, 9, 10, 11, 12, 14, 16, and 23 of the Technical Report.
6. I am independent of Scozinc Limited as defined by Section 1.5 of the Instrument.
7. I was involved with the property from 2007 to 2009 with another owner/operator during operations. I am not, nor have I ever been an employee of ScoZinc or Selwynn. I have read the Instrument and the section of the Technical Report that I am responsible for, and it has been prepared in compliance with the Instrument.
8. I have read the Instrument and the sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated 02 February 2018.

{Original Signed & Sealed}
Signature

Jason Baker, P. Eng.
Consultant
Stantec Consulting Ltd.

Stantec

CERTIFICATE OF AUTHOR

To: ScoZinc Mining Ltd. (“**ScoZinc**”)

I, Jennifer L. McPhail, M.Eng., P.Eng., of Fredericton, NB, do hereby certify that:

1. I am an Associate and Project Manager with Stantec Consulting Ltd. (“**Stantec**”), with a business mailing address at 845 Prospect Street, Fredericton, NB E3B 2T7.
2. This certificate applies to the Technical Report entitled “ScoZinc Mine – Preliminary Economic Assessment Update – Document No. RPT-17698” dated 02 February 2018.
3. I am a graduate of the University of New Brunswick, (B.Sc. 2004, M.Eng., 2006). I am a member in good standing of the Association of Professional Engineers and Geoscientists of New Brunswick, licence number M7036. My relevant experience is approximately 10 years of experience in environmental assessments within the Atlantic provinces for a wide range of projects including several in the mining industry. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
4. I have not personally inspected the Property.
5. I am responsible for Section 20 of the Technical Report.
6. I am independent of Scozinc Limited as defined by Section 1.5 of the Instrument.
7. I have no prior involvement with the Property that is the subject of the Technical Report.
8. I have read the Instrument and the section of the Technical Report that I am responsible for, and it has been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated 02 February 2018.

{Original Signed and Sealed}
Signature

Jennifer L. McPhail, M. Eng., P. Eng.
Associate
Stantec Consulting Ltd.

CERTIFICATE OF AUTHOR

To: ScoZinc Mining Ltd. (“**ScoZinc**”)

I, Kim Trapani, P.Eng., of Sudbury, ON, do hereby certify that:

1. I am an Intermediate Engineer employed with Stantec Consulting Ltd. (“**Stantec**”), with a business mailing address at 1-1760 Regent Street, Sudbury, Ontario P3E 3Z8.
2. This certificate applies to the Technical Report entitled “ScoZinc Mine – Preliminary Economic Assessment Update – Document No. RPT-17698” dated 02 February 2018.
3. I am a graduate of the University of Exeter, (BSc, 2010). I am a member in good standing of the Association of Professional Engineers of Ontario, licence number 100212325. My relevant experience is approximately 7 years within the mining industry, of which 2 years was focussed on techno-economic assessments. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
4. I am responsible for Sections 21 and 22 of the Technical Report.
5. I have not personally inspected the Property.
6. I am independent of Scozinc Limited as defined by Section 1.5 of the Instrument.
7. I have no prior involvement with the Property that is the subject of the Technical Report.
8. I have read the Instrument and the sections of the Technical Report that I am responsible for, and it has been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated 02 February 2018.

{Original Signed and Sealed}
Signature

Kim Trapani, P. Eng.
Intermediate Engineer
Stantec Consulting Ltd.

CERTIFICATE OF AUTHOR

To: ScoZinc Mining Ltd. ("**ScoZinc**")

I, Michael Romaniuk, P.Eng., of Sudbury, ON, do hereby certify that:

1. I am a Senior Associate employed with Stantec Consulting Ltd. ("**Stantec**"), with a business mailing address at 1-1760 Regent Street, Sudbury, Ontario P3E 3Z8.
2. This certificate applies to the Technical Report entitled "ScoZinc Mine – Preliminary Economic Assessment Update – Document No. RPT-17698" dated 02 February 2018.
3. I am a graduate of the University of Toronto , (BASc, 1987). I am a member in good standing of Professional Engineers Ontario , licence number 90379926. My relevant experience is approximately 30 years of project and operational experience on a wide range of projects. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
4. I am responsible for Sections 1, 2, 3, 4, 5, 6, 13, 17, 19, 24, 25 and 27 of the Technical Report.
5. I have personally inspected the Property.
6. I am independent of Scozinc Limited as defined by Section 1.5 of the Instrument.
7. I have no prior involvement with the Property that is the subject of the Technical Report.
8. I have read the Instrument and the sections of the Technical Report that I am responsible for, and it has been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated 02 February 2018.

{Original Signed and Sealed}
Signature

Michael Romaniuk, P. Eng.
Senior Associate
Stantec Consulting Ltd.

CERTIFICATE OF AUTHOR

To: ScoZinc Mining Ltd. ("**ScoZinc**")

I, Stephen J. Oaks, P.Eng., of Sudbury, ON, do hereby certify that:

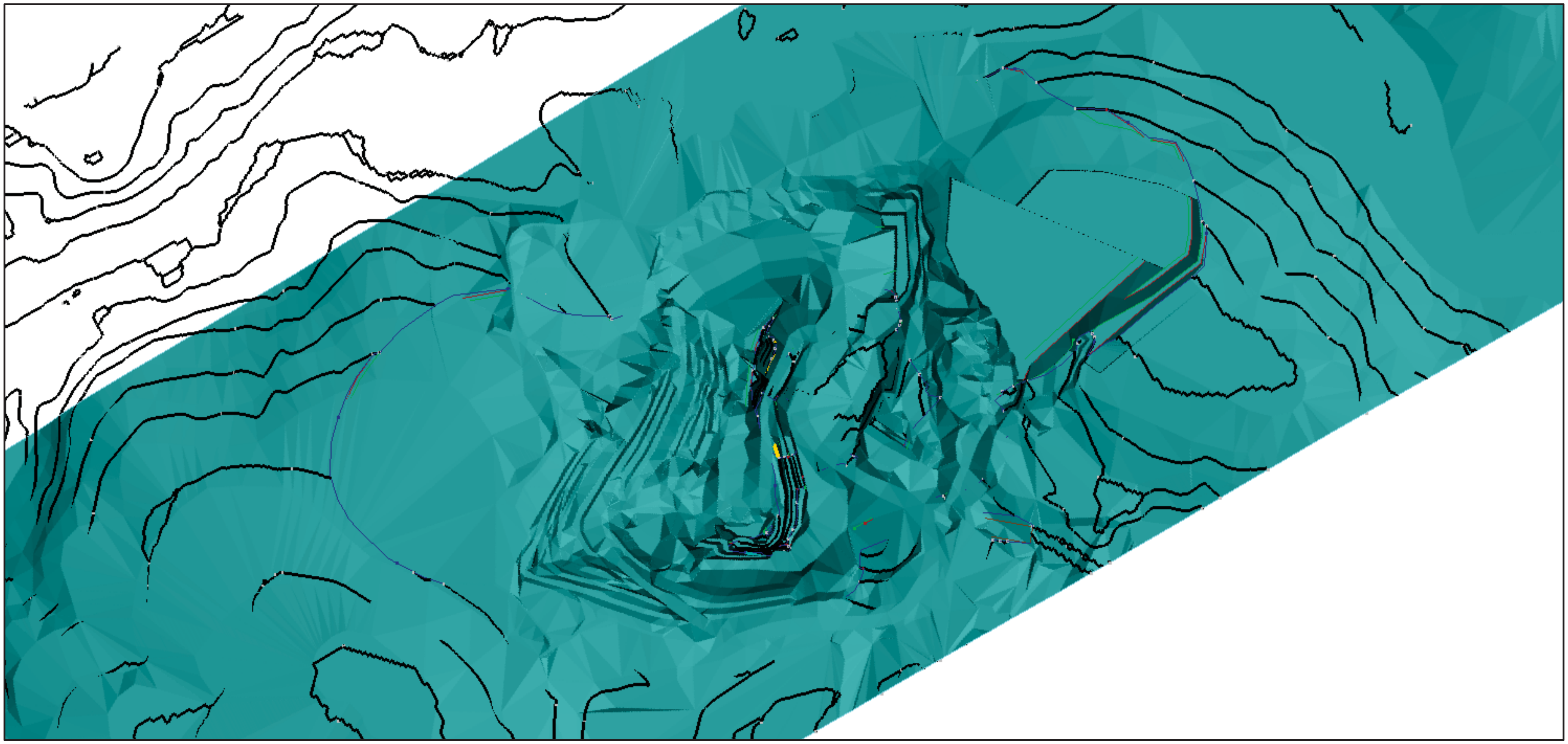
1. I am a Project Specialist employed with Stantec Consulting Ltd. ("**Stantec**"), with a business mailing address at 1-1760 Regent Street, Sudbury, Ontario P3E 3Z8.
2. This certificate applies to the Technical Report entitled "ScoZinc Mine – Preliminary Economic Assessment Update – Document No. RPT-17698" dated 02 February 2018.
3. I am a graduate of the University of Waterloo, (BASc, 1976). I am a member in good standing of the Association of Professional Engineers of Ontario, licence number 34450015. My relevant experience is approximately 25 years of infrastructure experience within the mining, exploration and manufacturing industry. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
4. I am responsible for Section 18 of the Technical Report.
5. I have personally inspected the Property.
6. I am independent of Scozinc Limited as defined by Section 1.5 of the Instrument.
7. I have no prior involvement with the Property that is the subject of the Technical Report.
8. I have read the Instrument and the sections of the Technical Report that I am responsible for, and it has been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated 02 February 2018.

{Original Signed and Sealed}
Signature

Stephen J. Oaks, P. Eng.
Project Specialist
Stantec Consulting Ltd.

Appendix 1
Figures from June 2013 ScoZinc Mine Preliminary Economic Assessment Update



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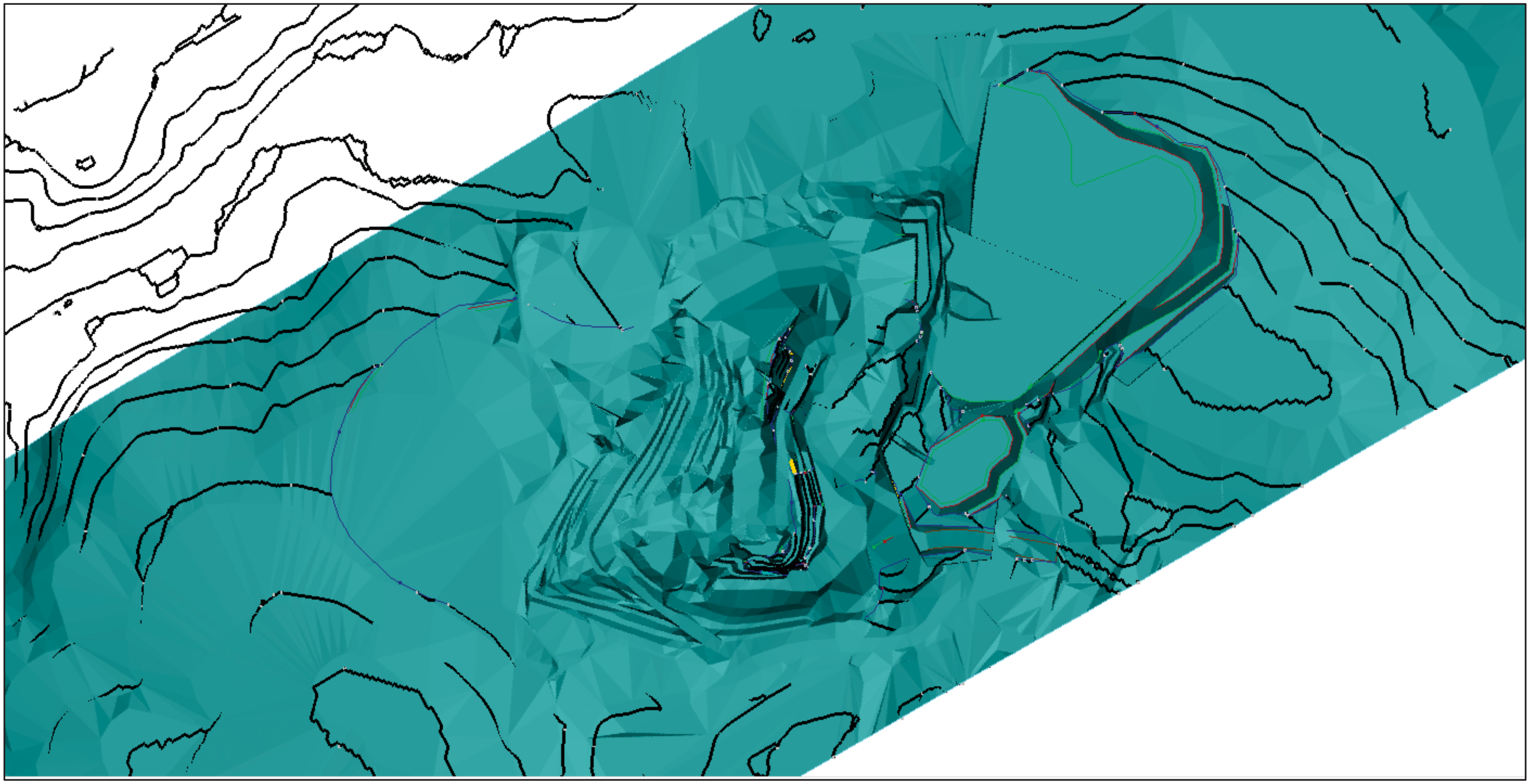

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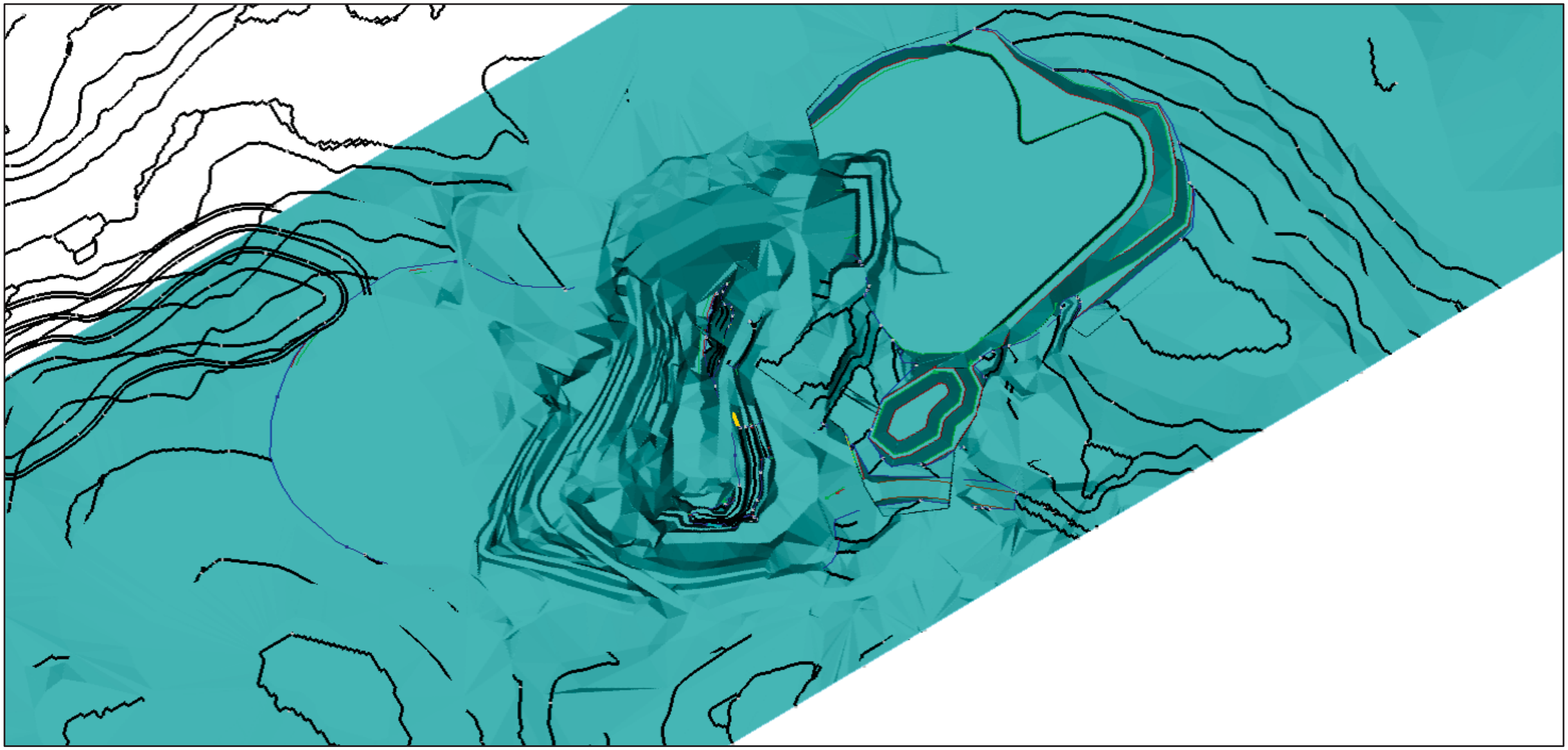

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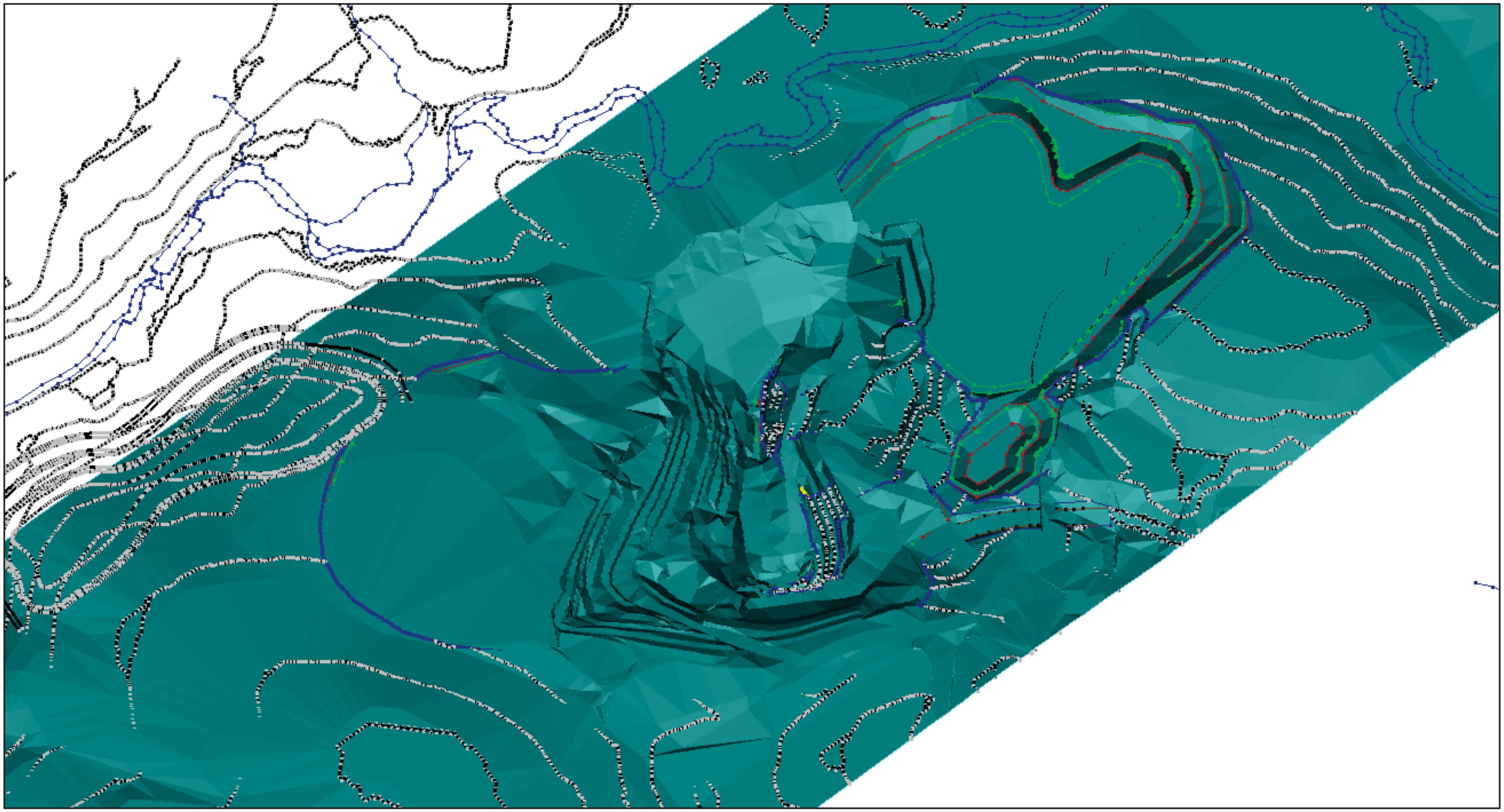

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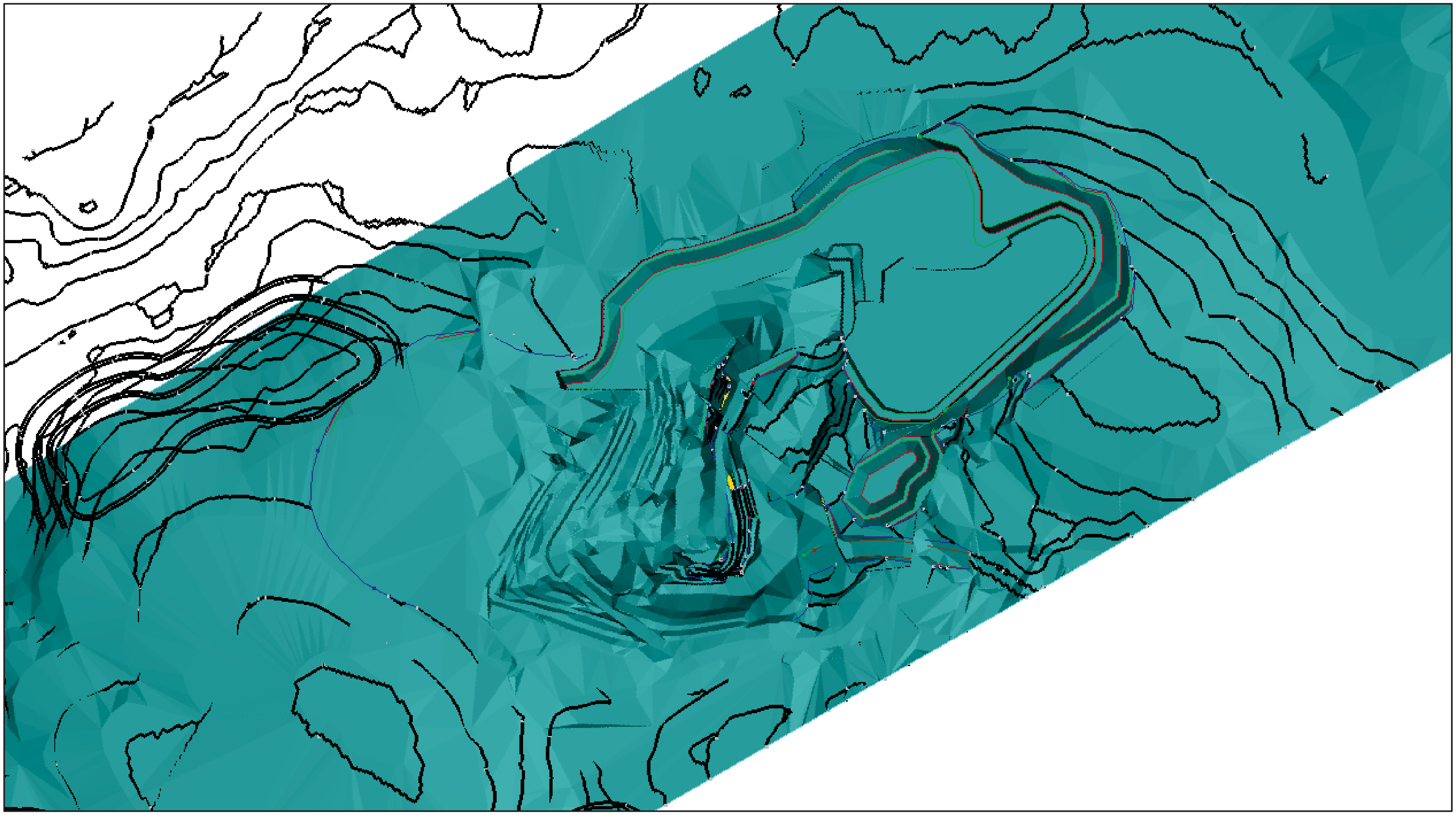

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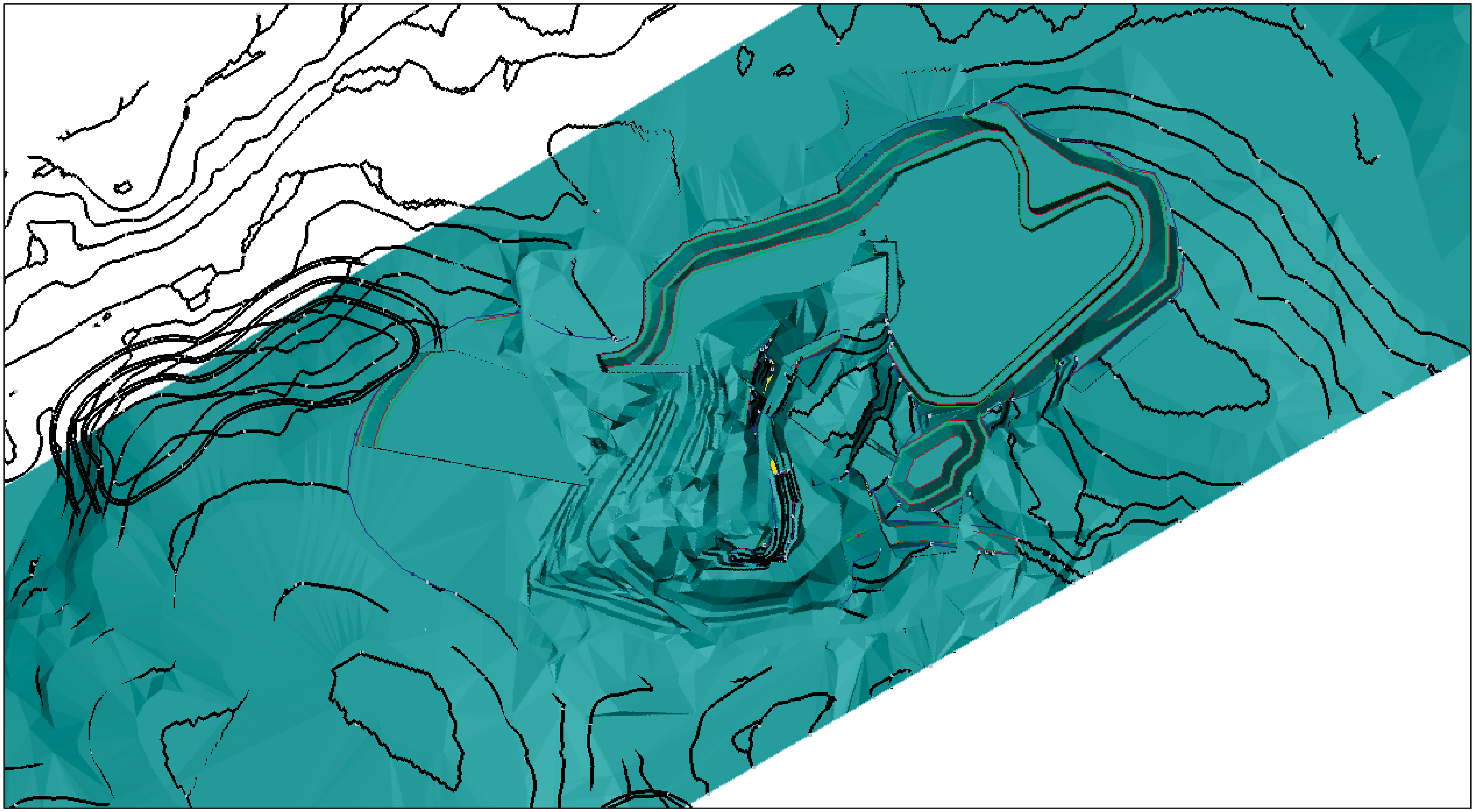

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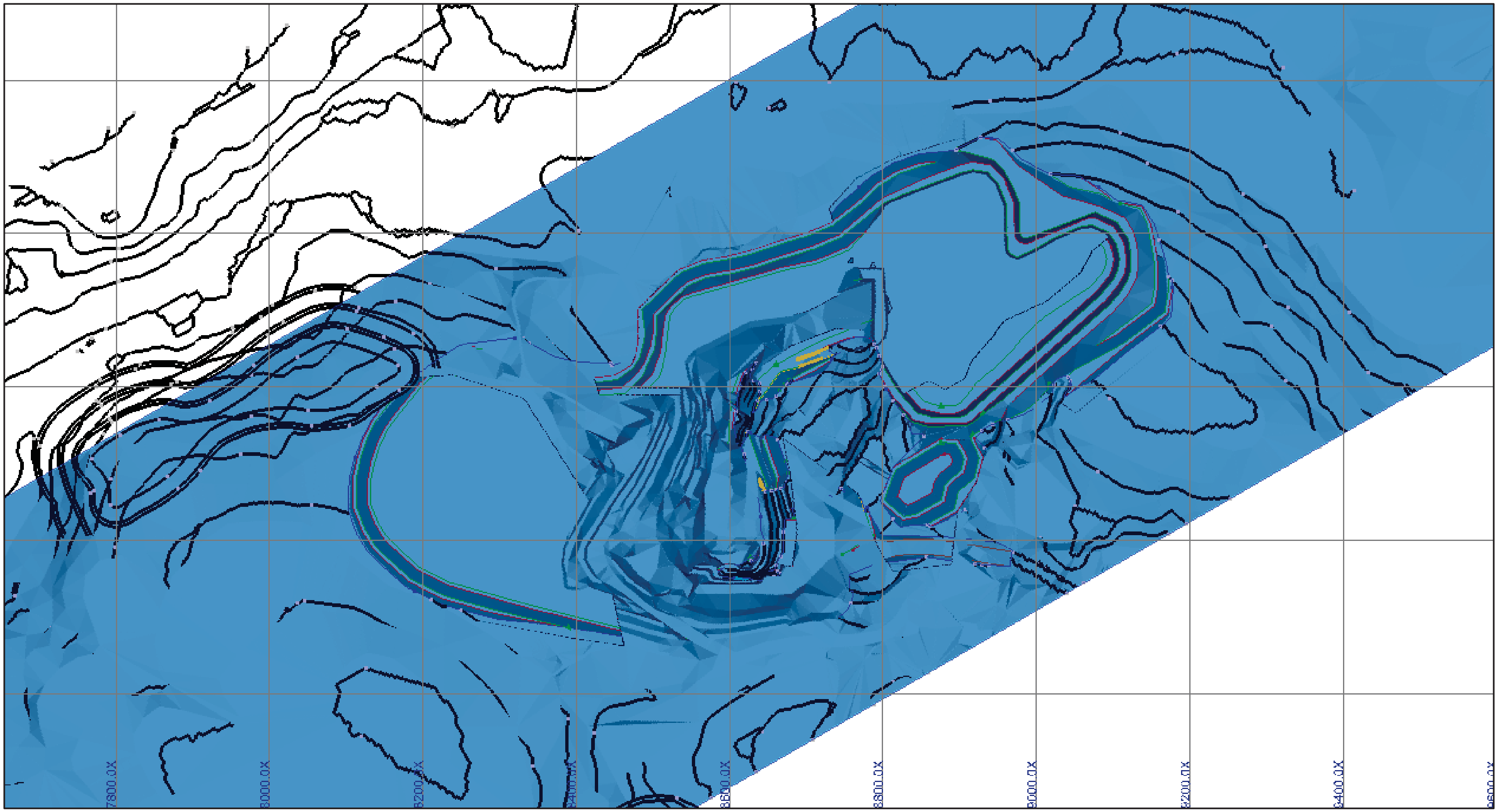

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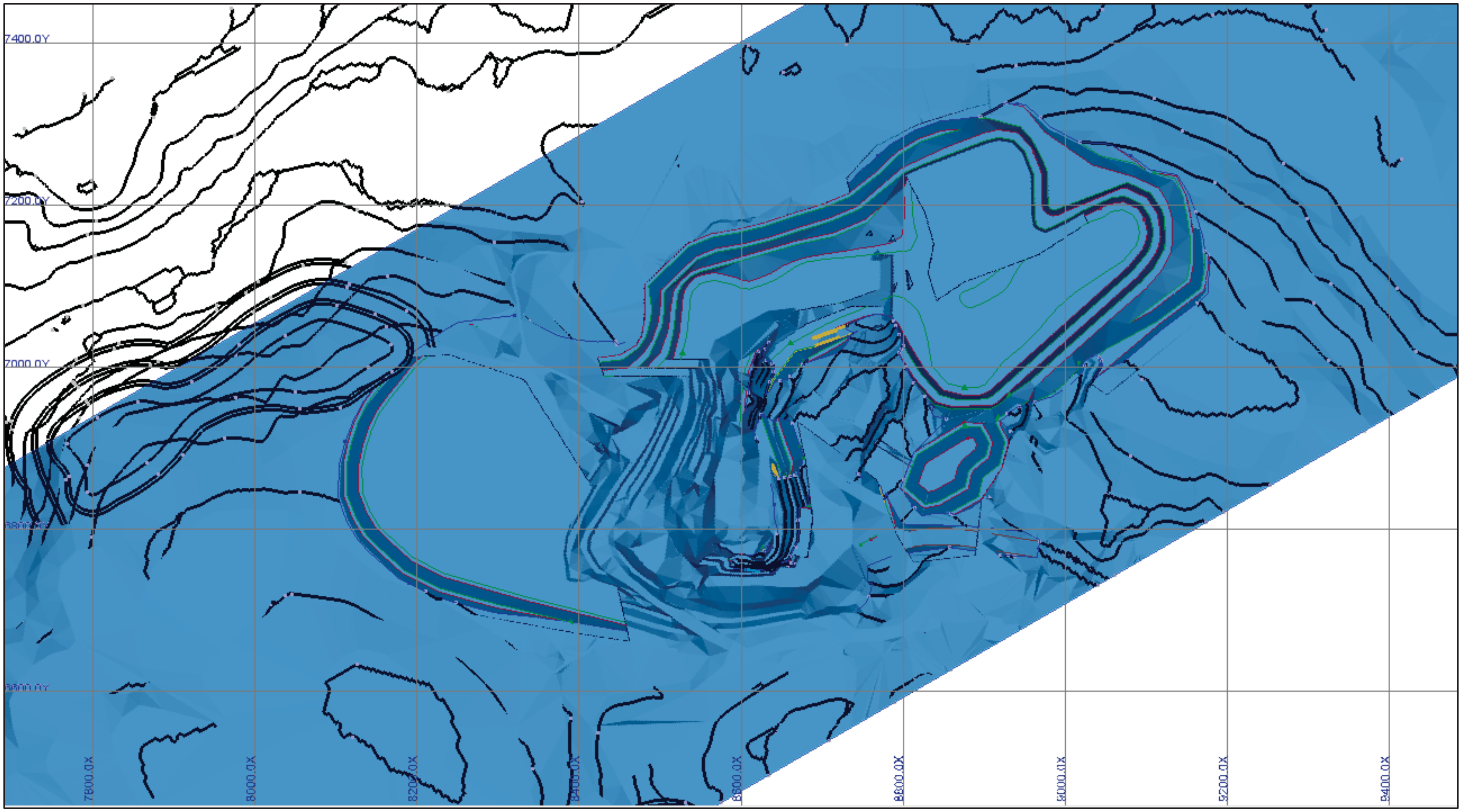

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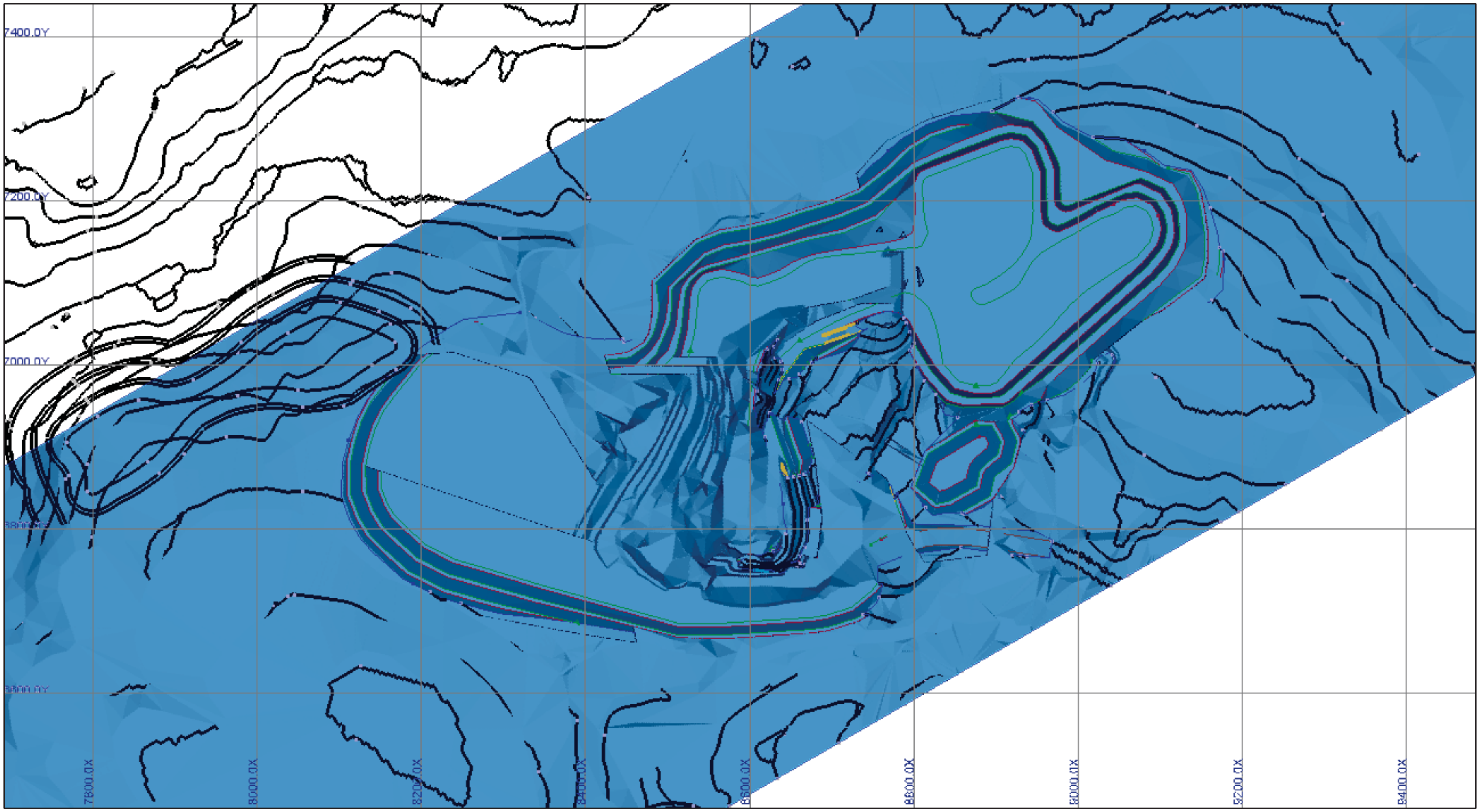

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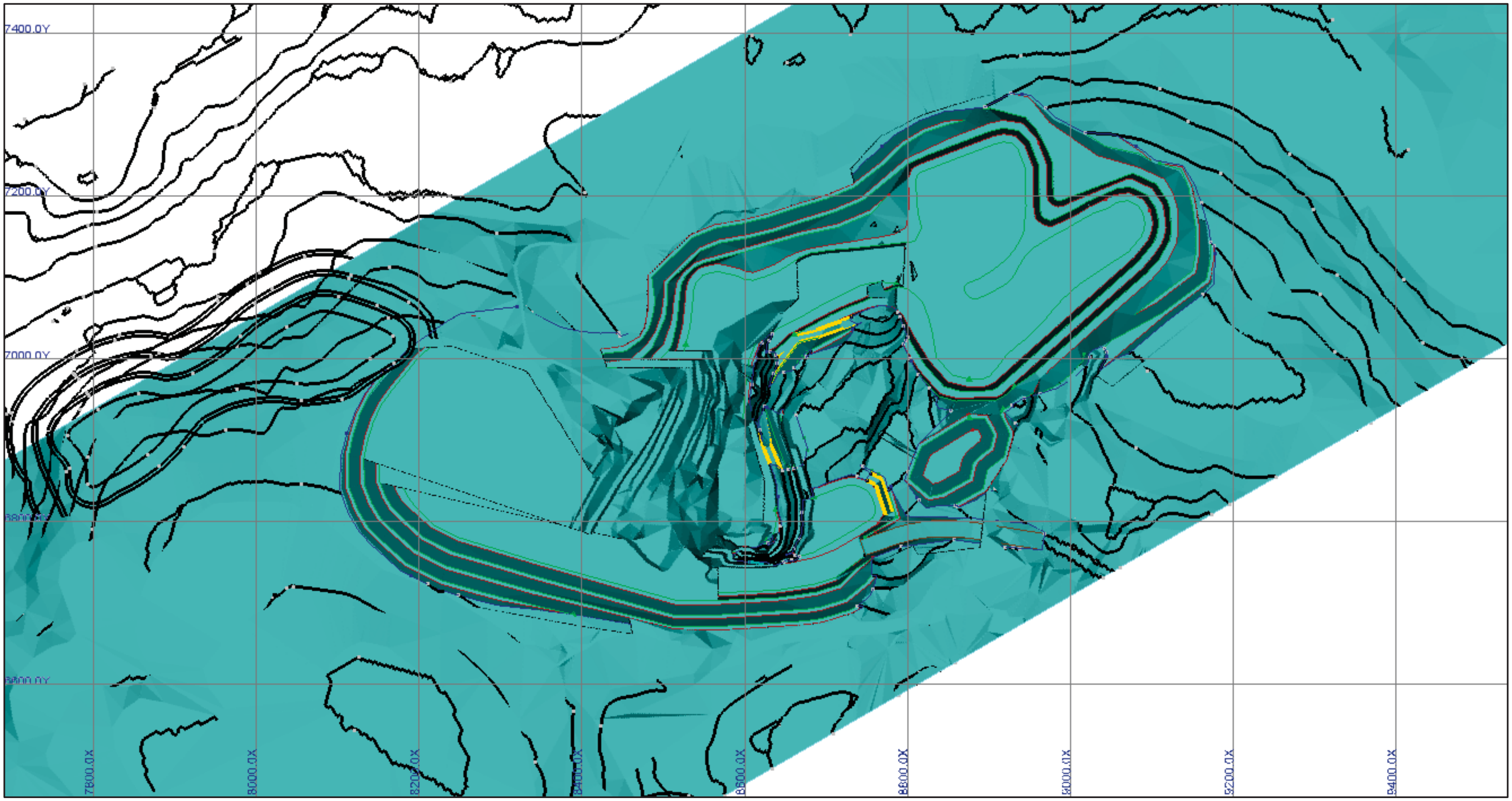

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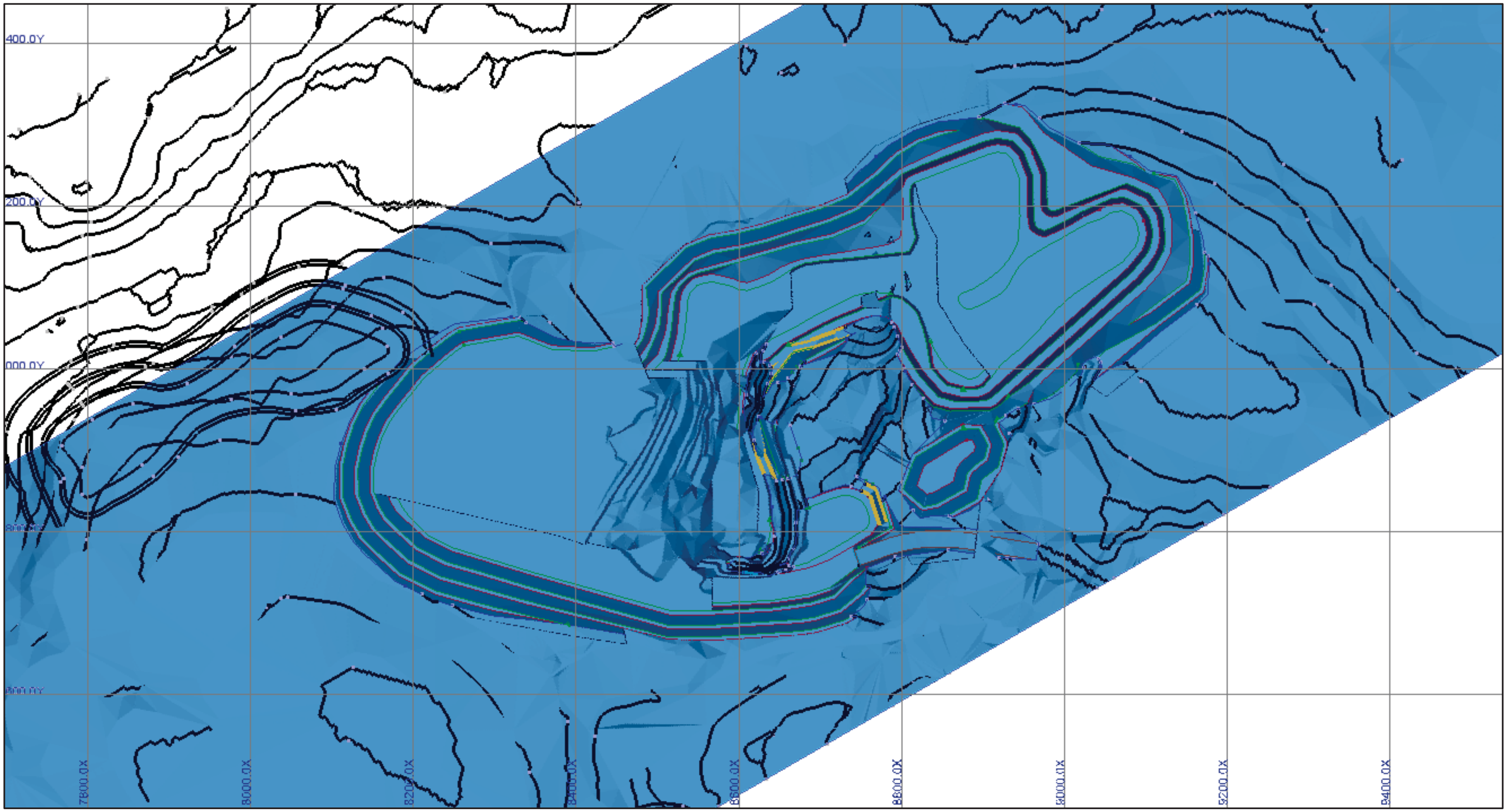

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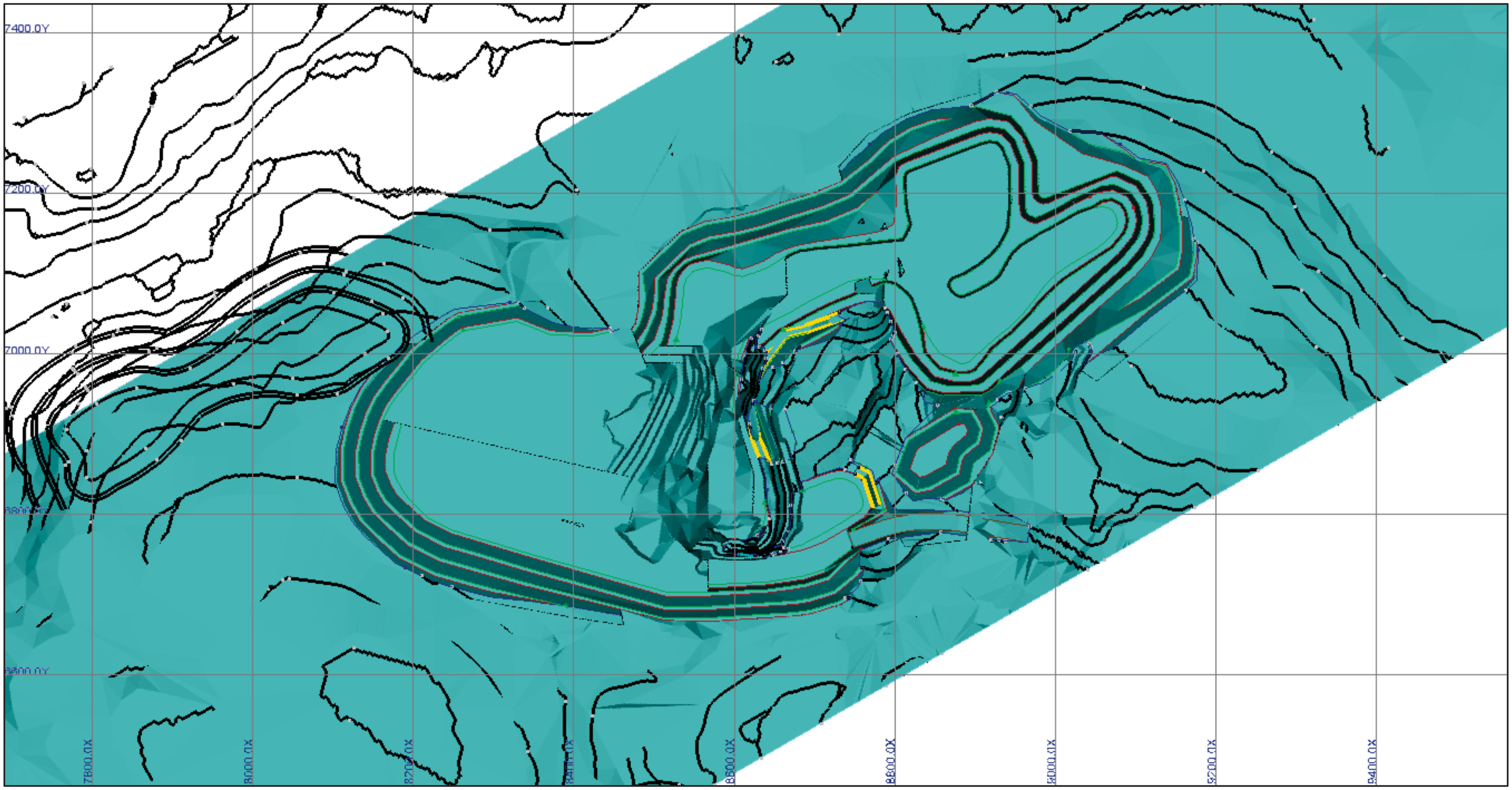

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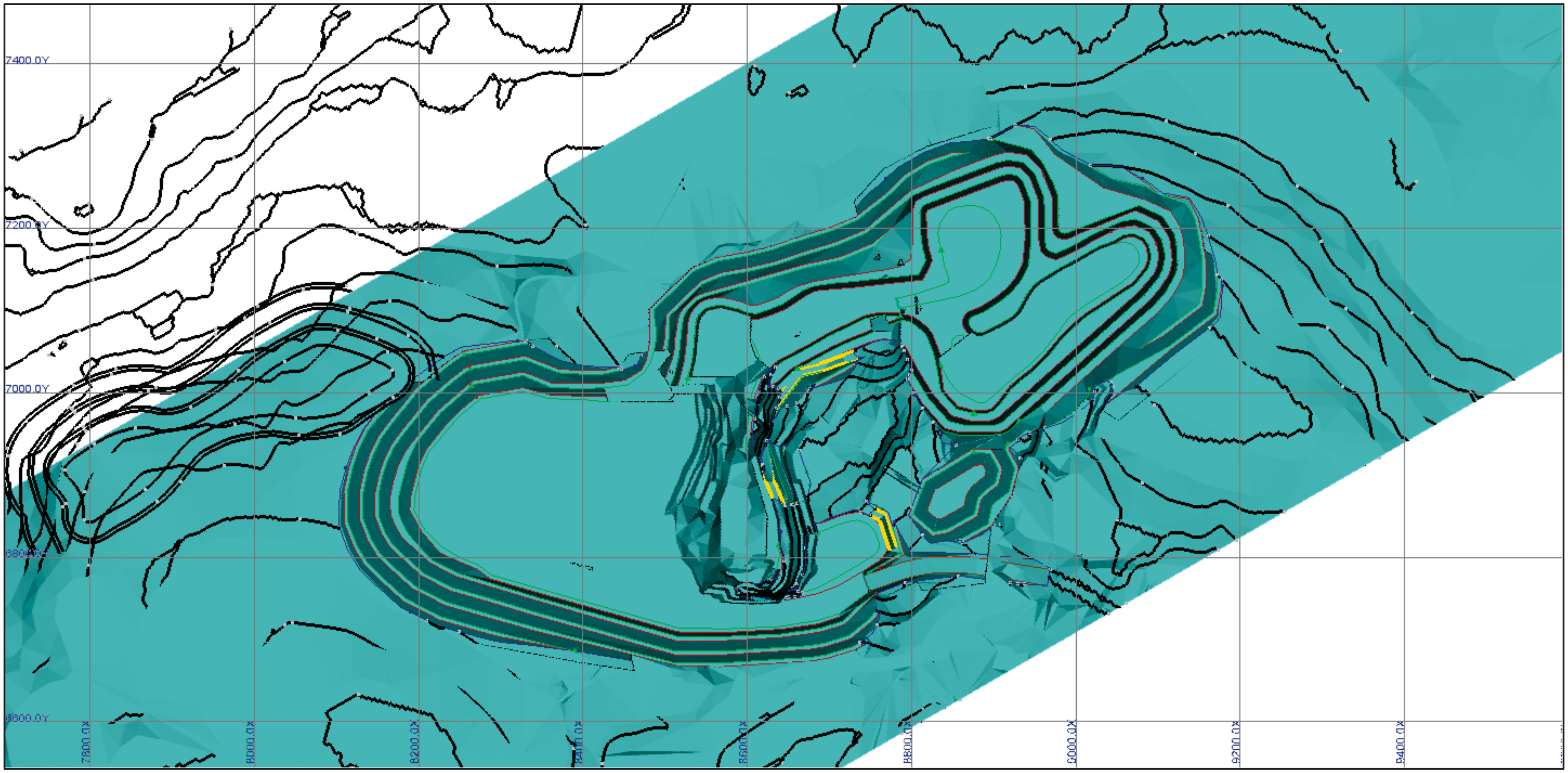

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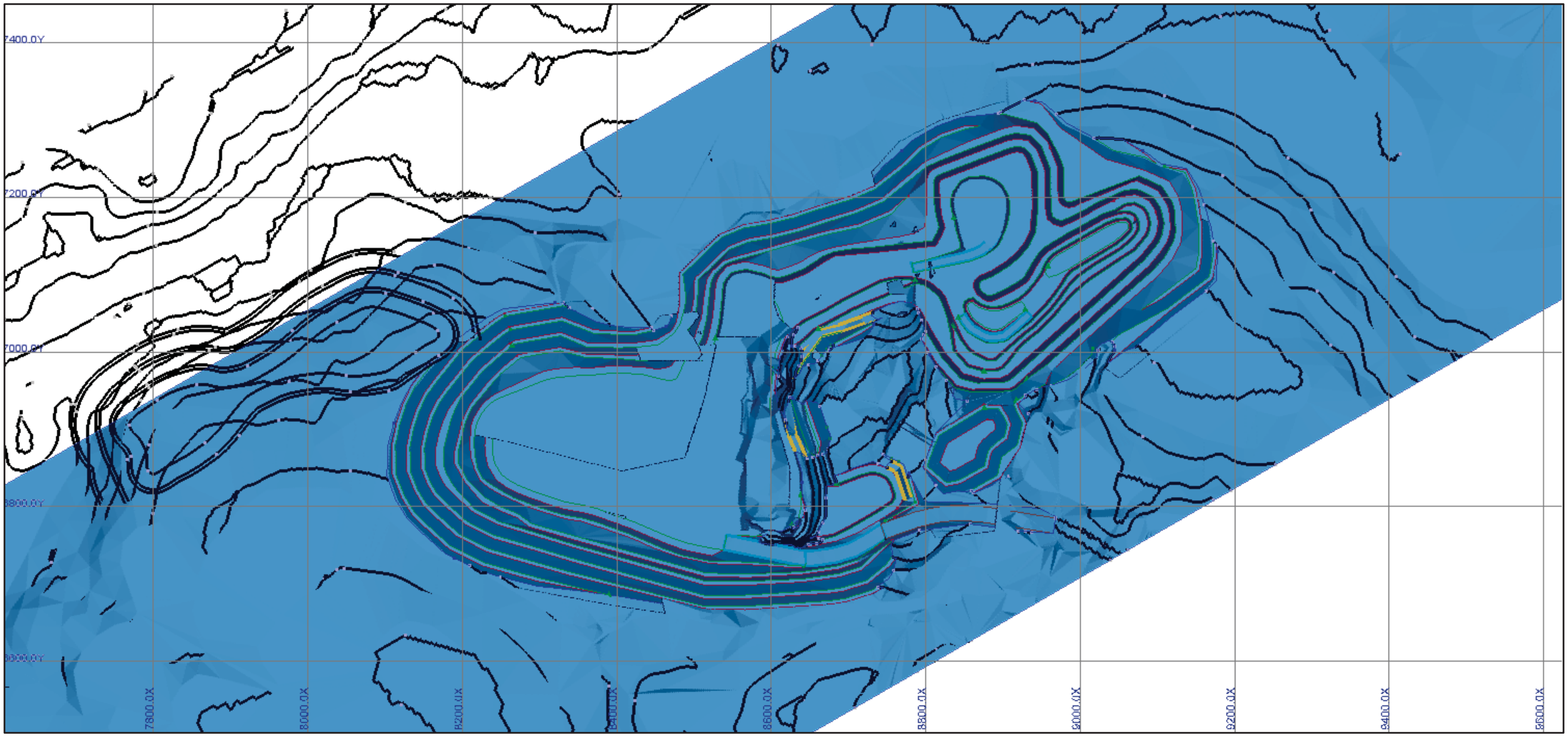

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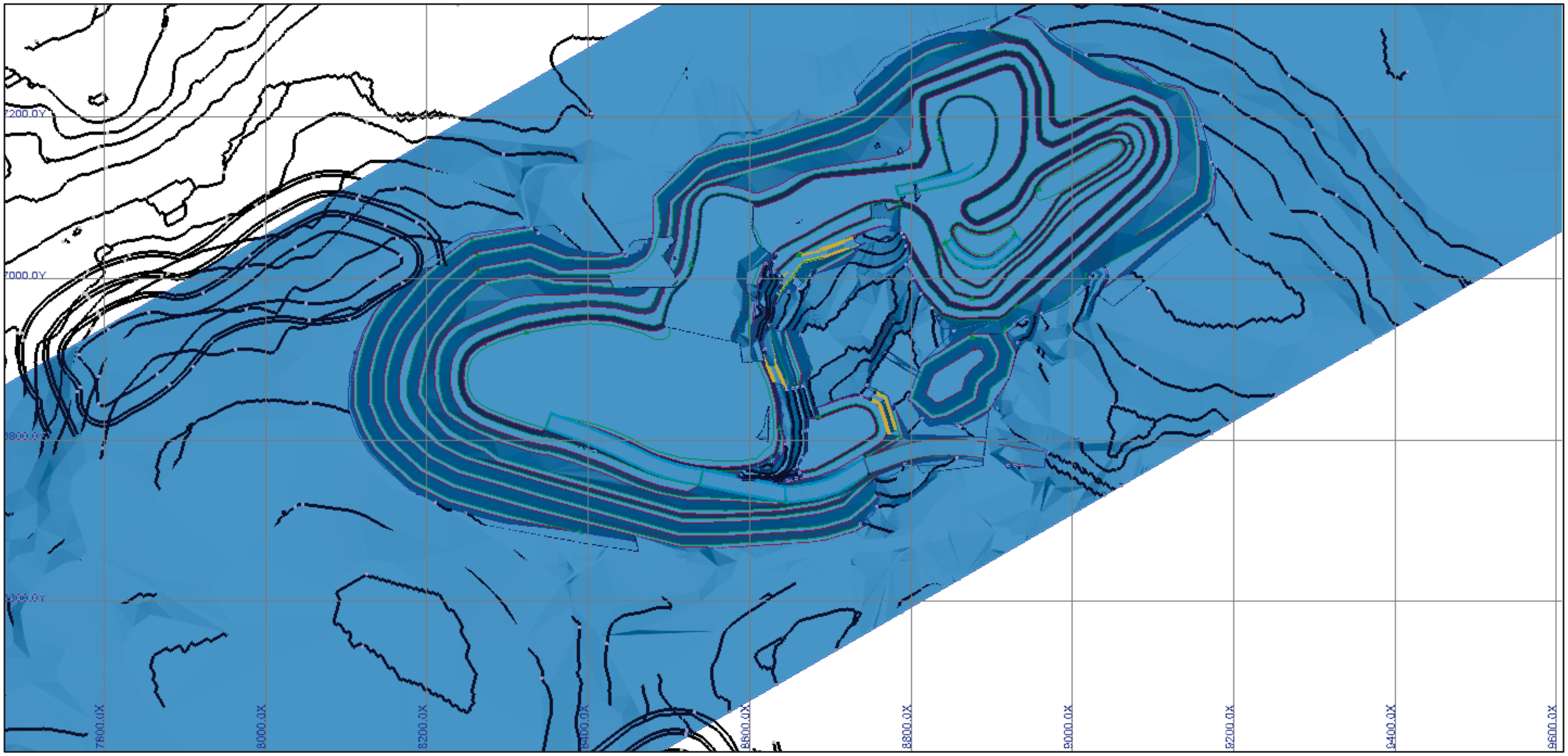

Figure 15 - Month 21 Cut

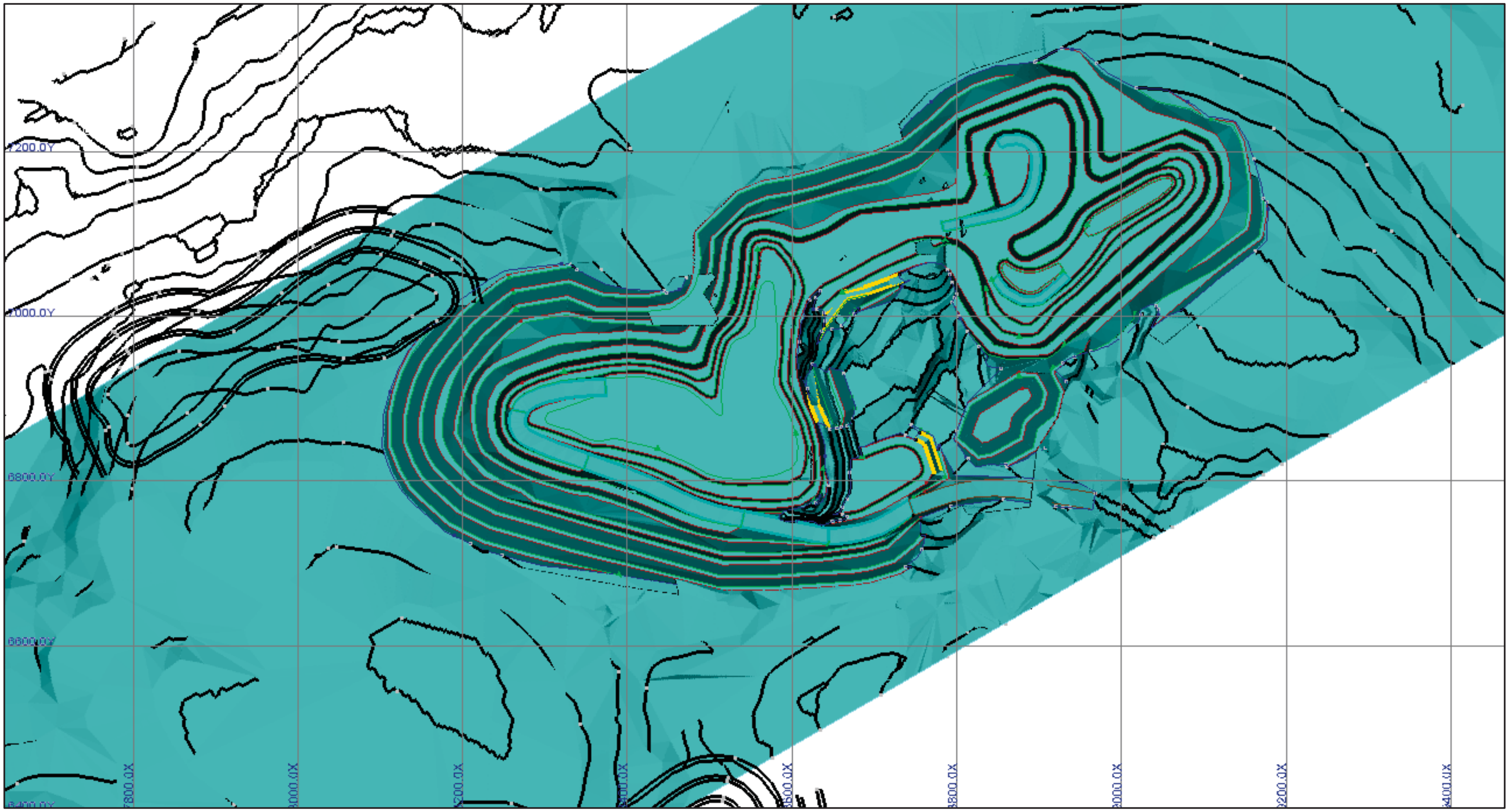

Figure 16 - Month 24 Cut

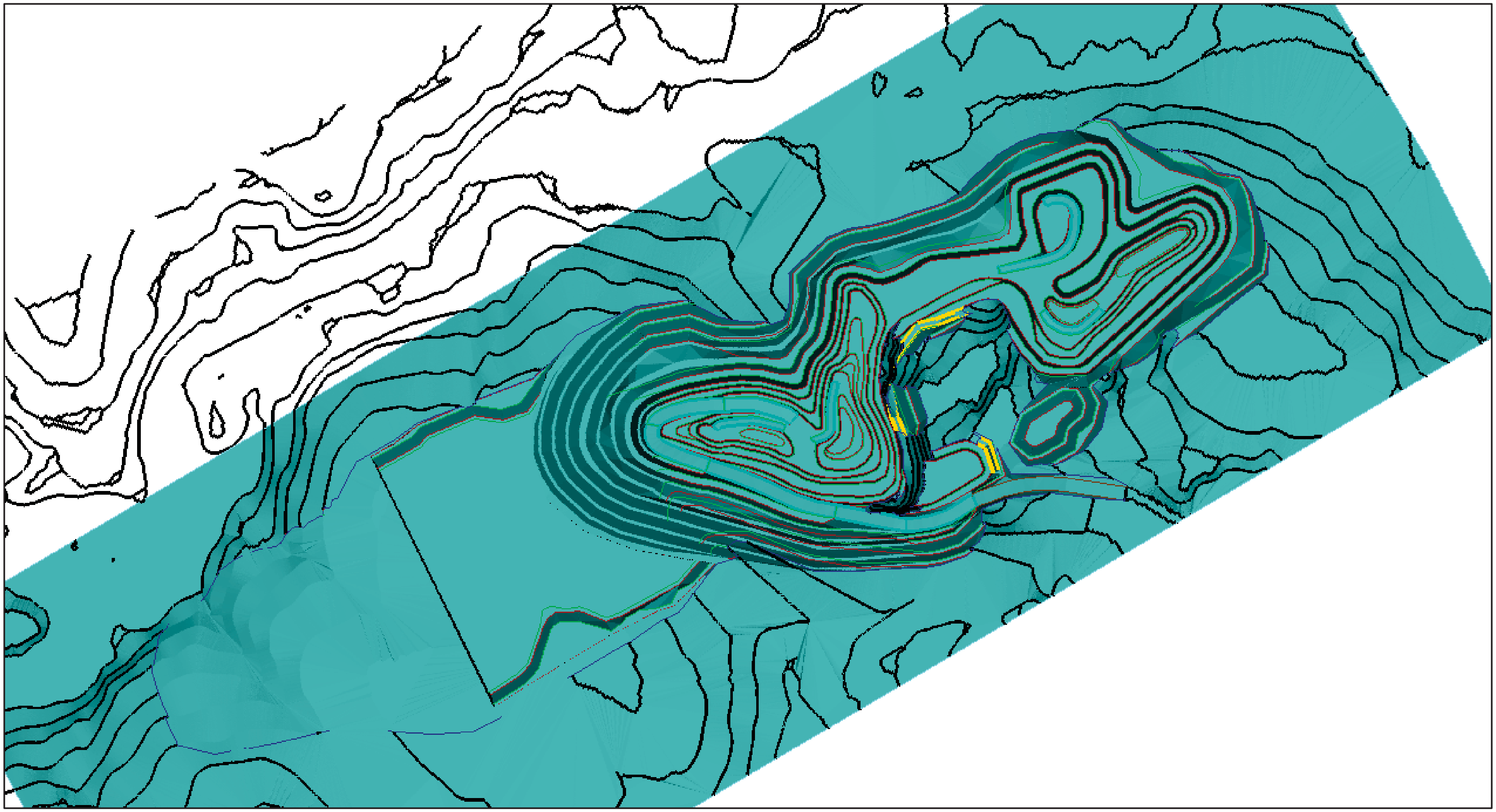

Figure 17 - Month 27 Cut

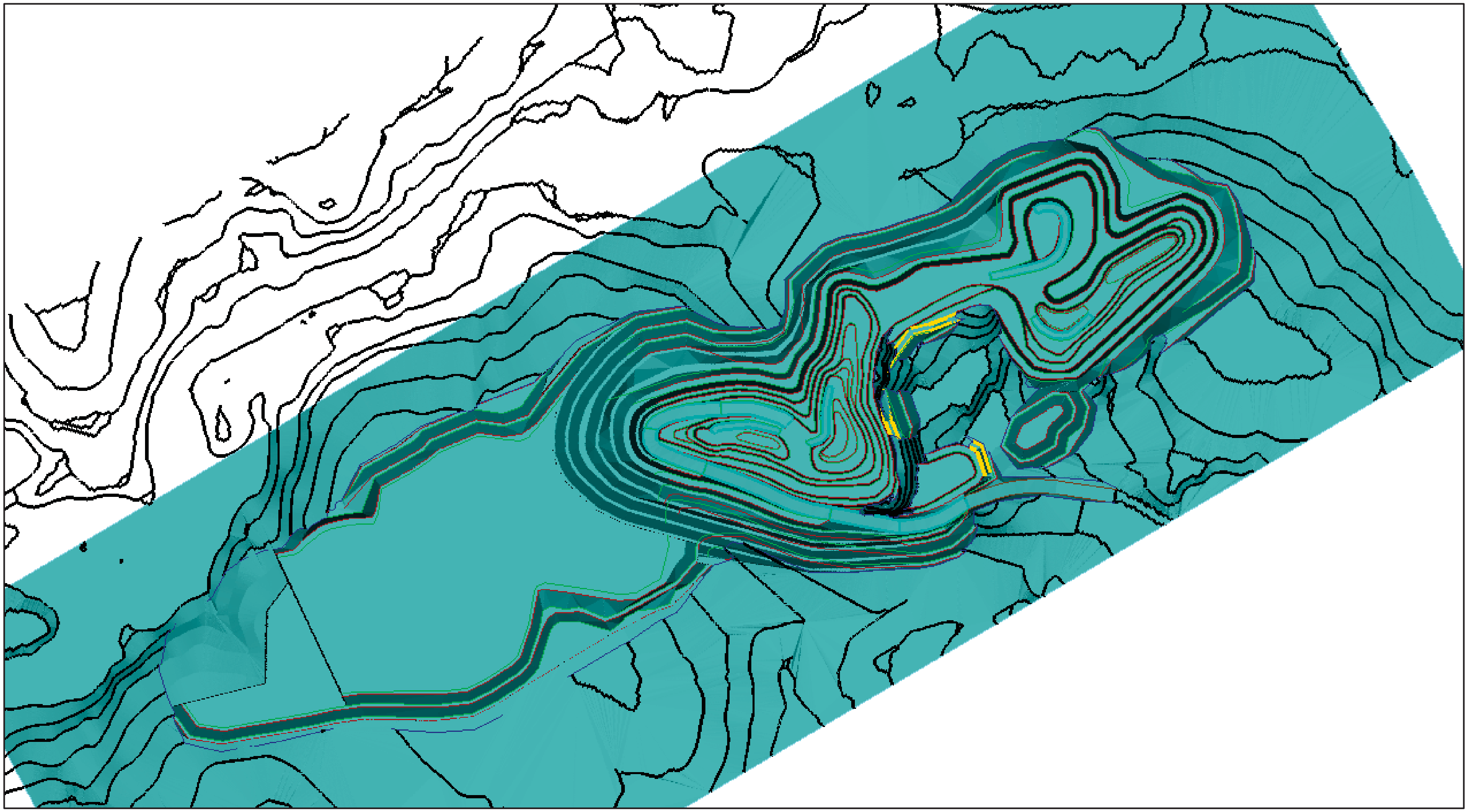

Figure 18 - Month 30 Cut

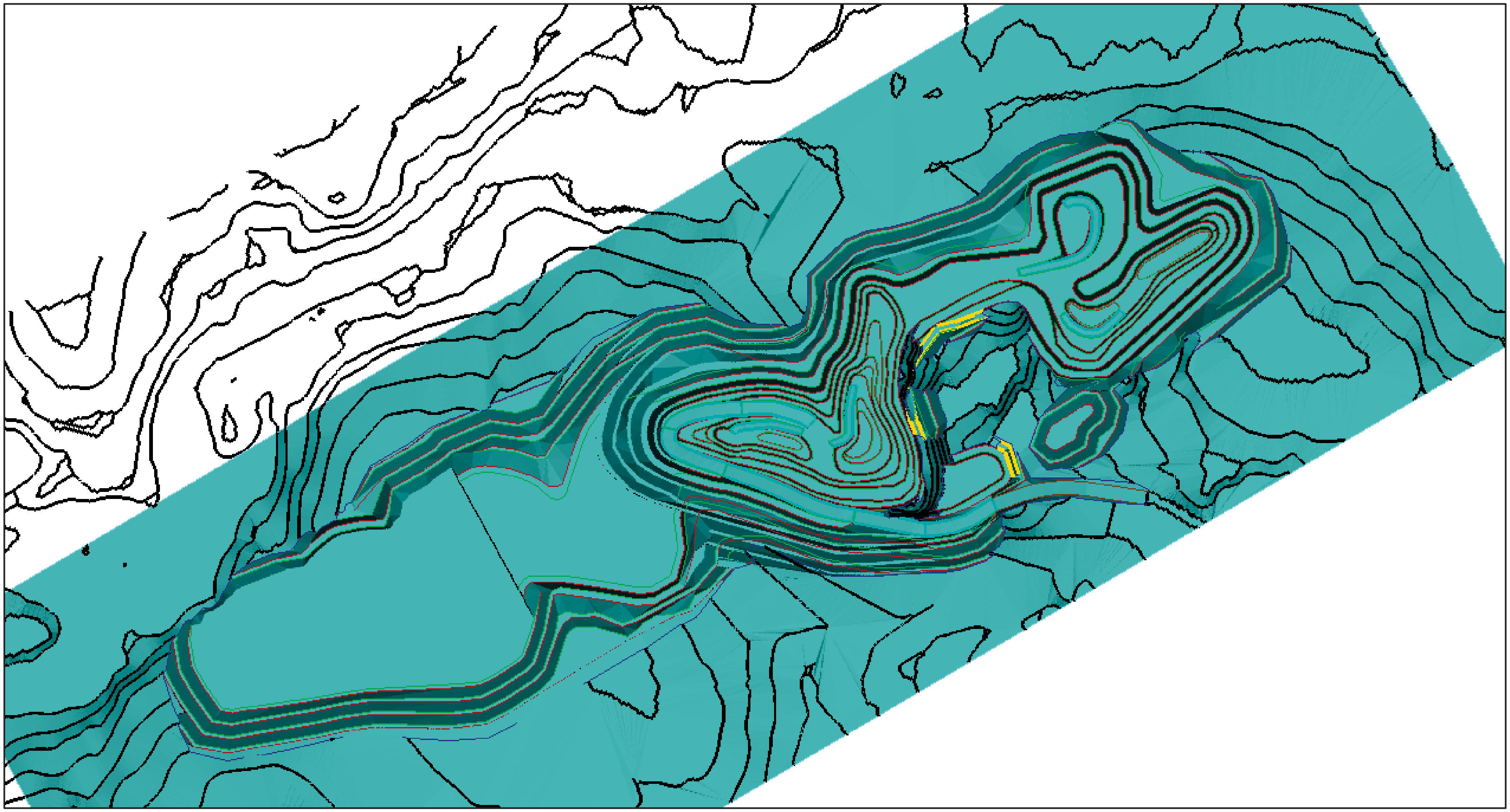

Figure 19 - Month 33 Cut

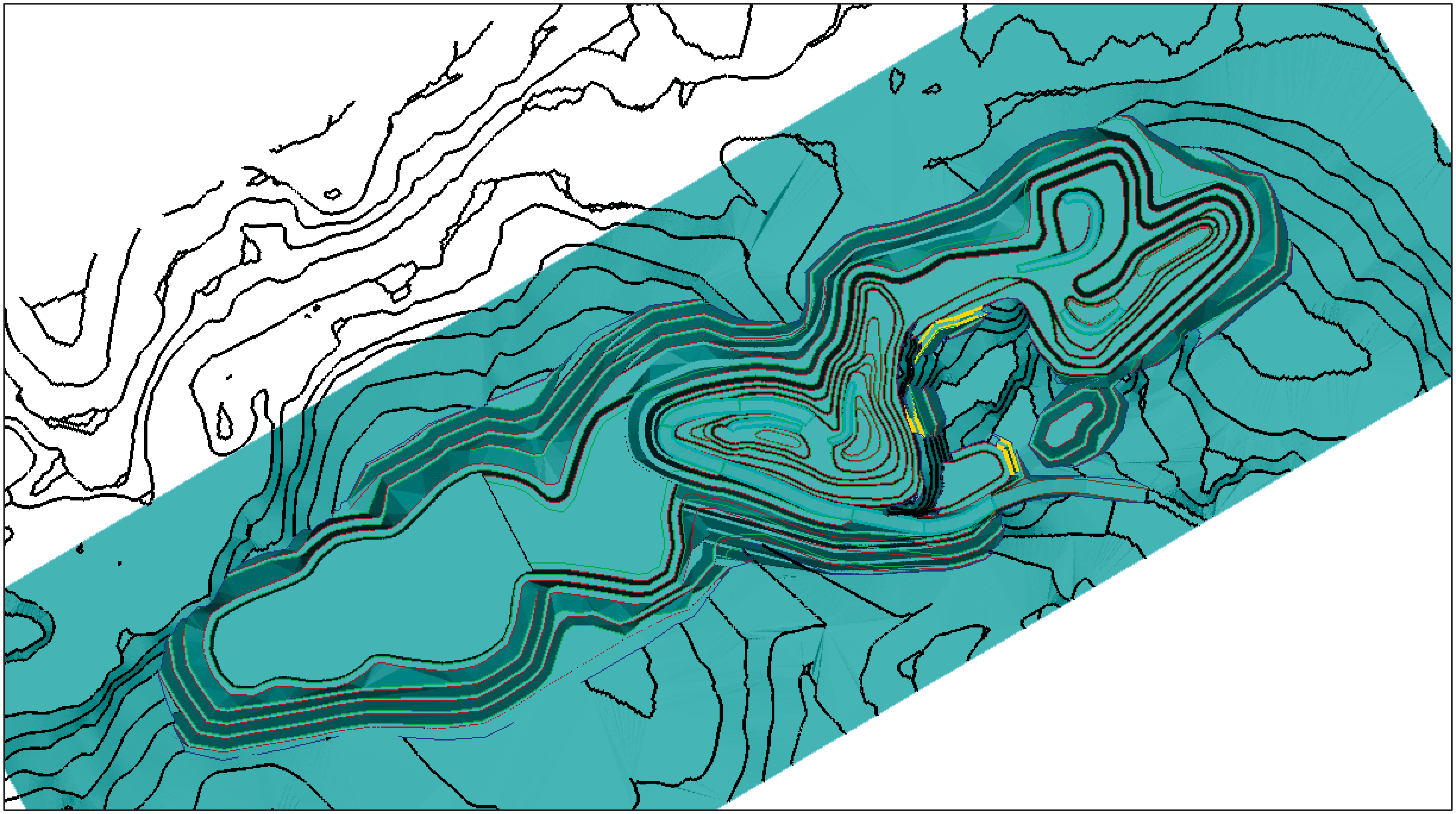

Figure 20 - Month 36 Cut

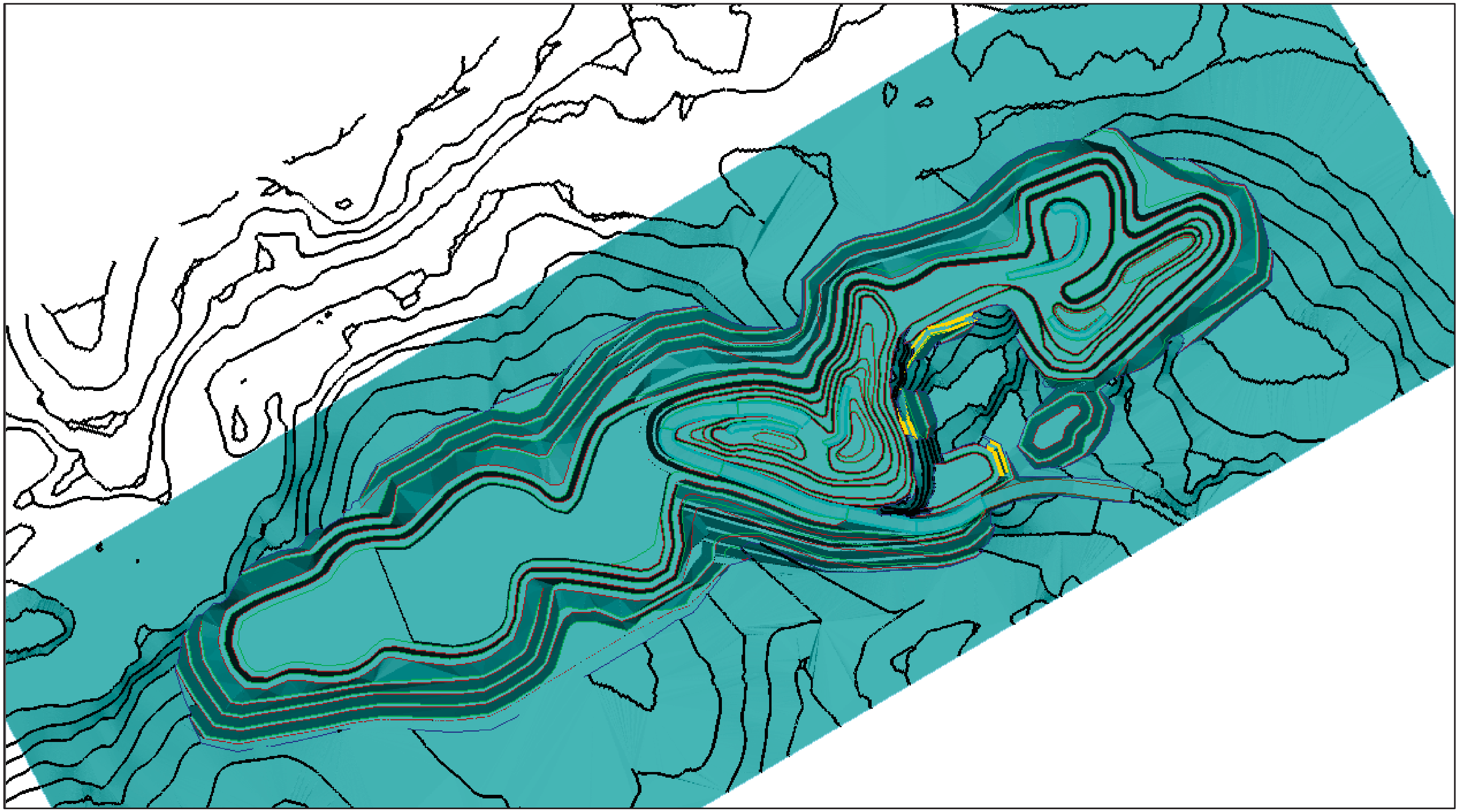

Figure 21 - Month 39 Cut

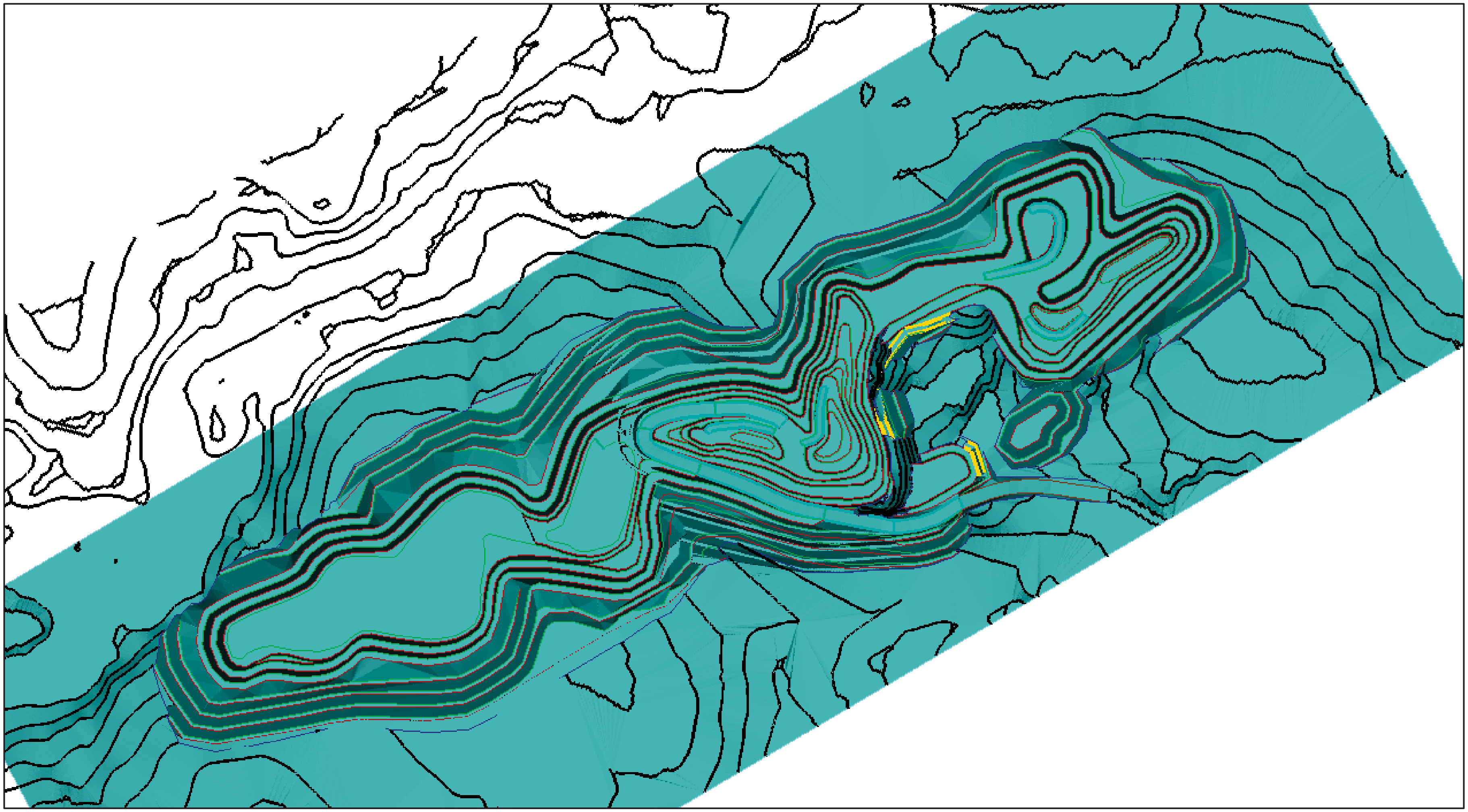

Figure 22 - Month 42 Cut

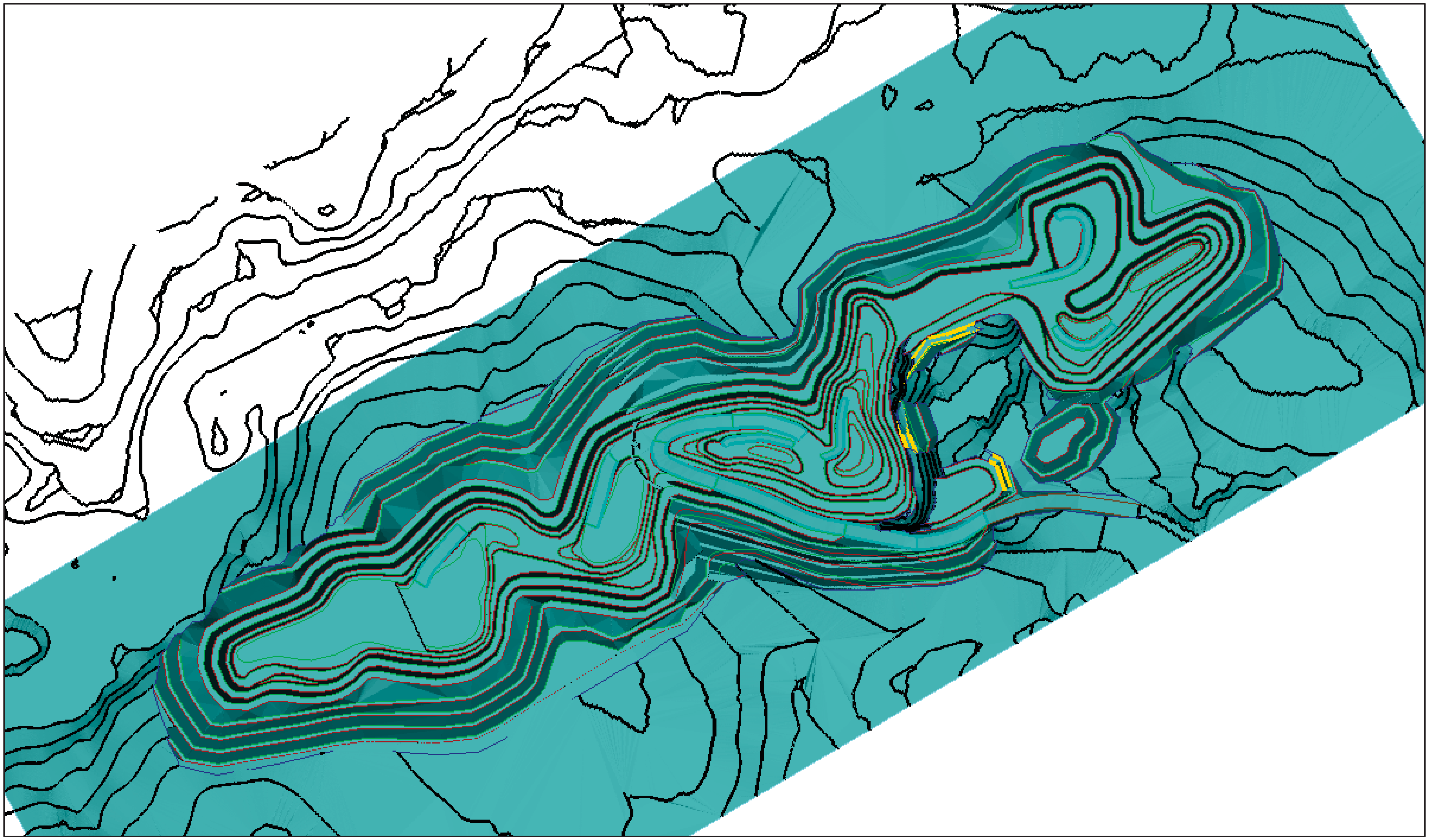

Figure 23 - Month 45 Cut

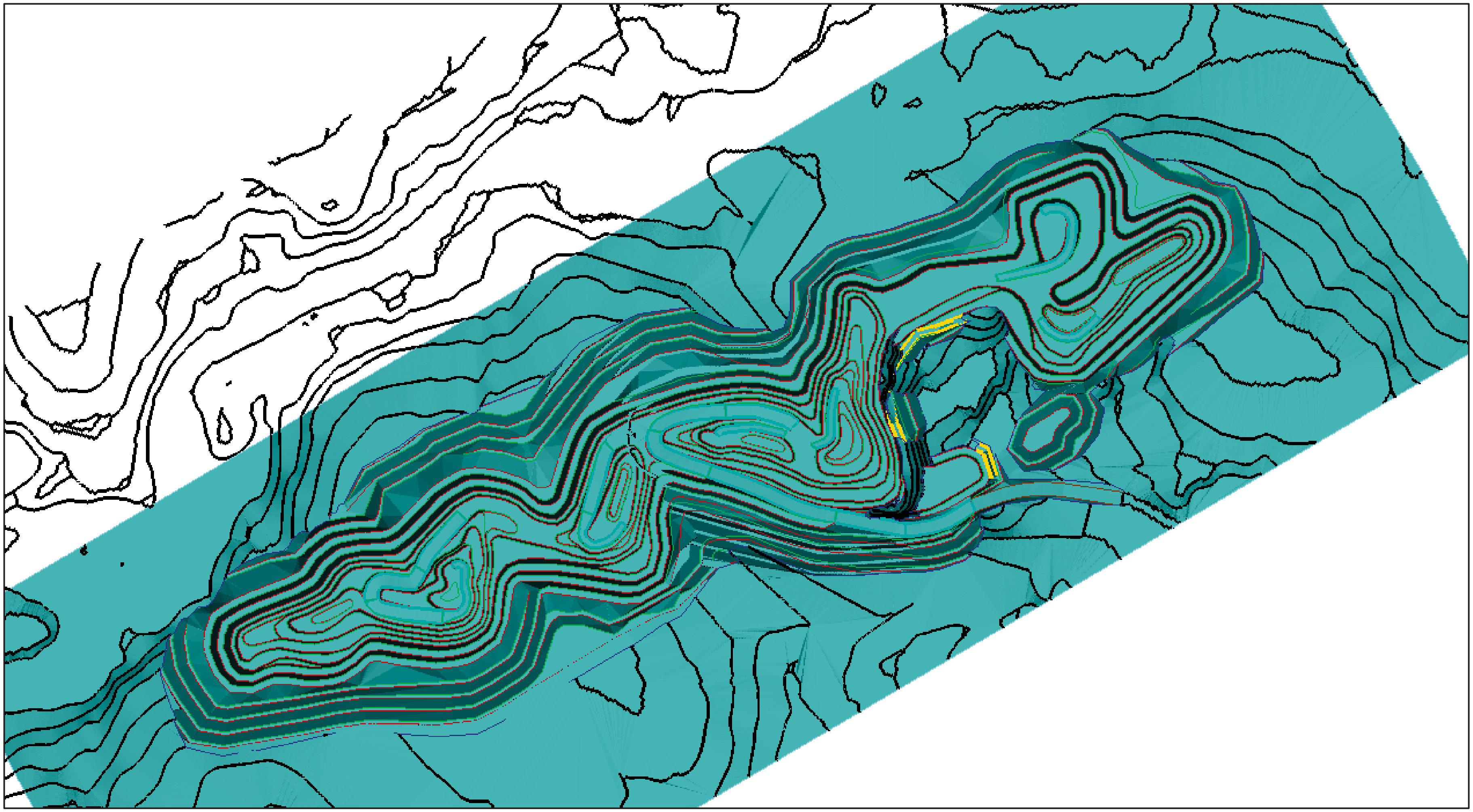

Figure 24 - Month 48 Cut

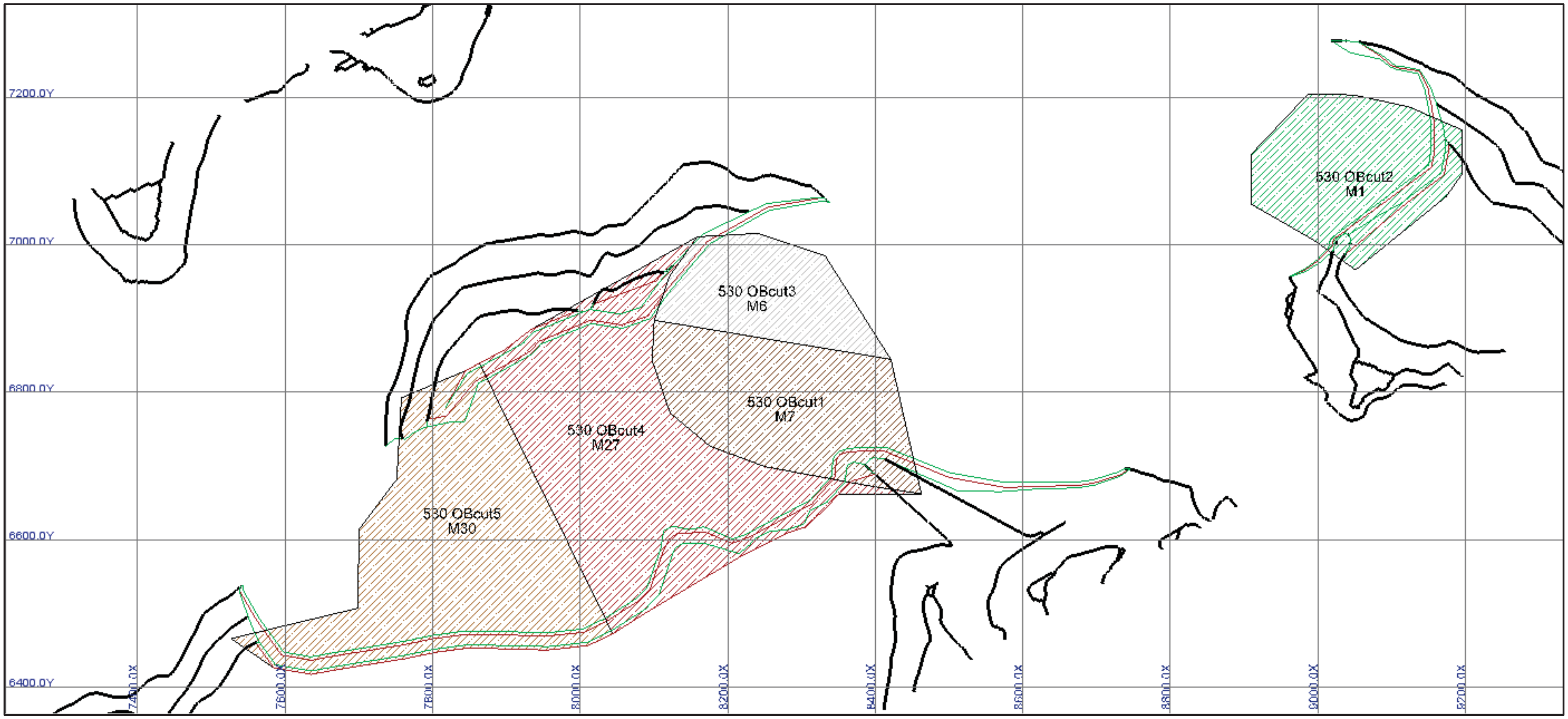

Figure 25 - 530 Bench Cut Showing Historic UG Workings (YR1-YR4)

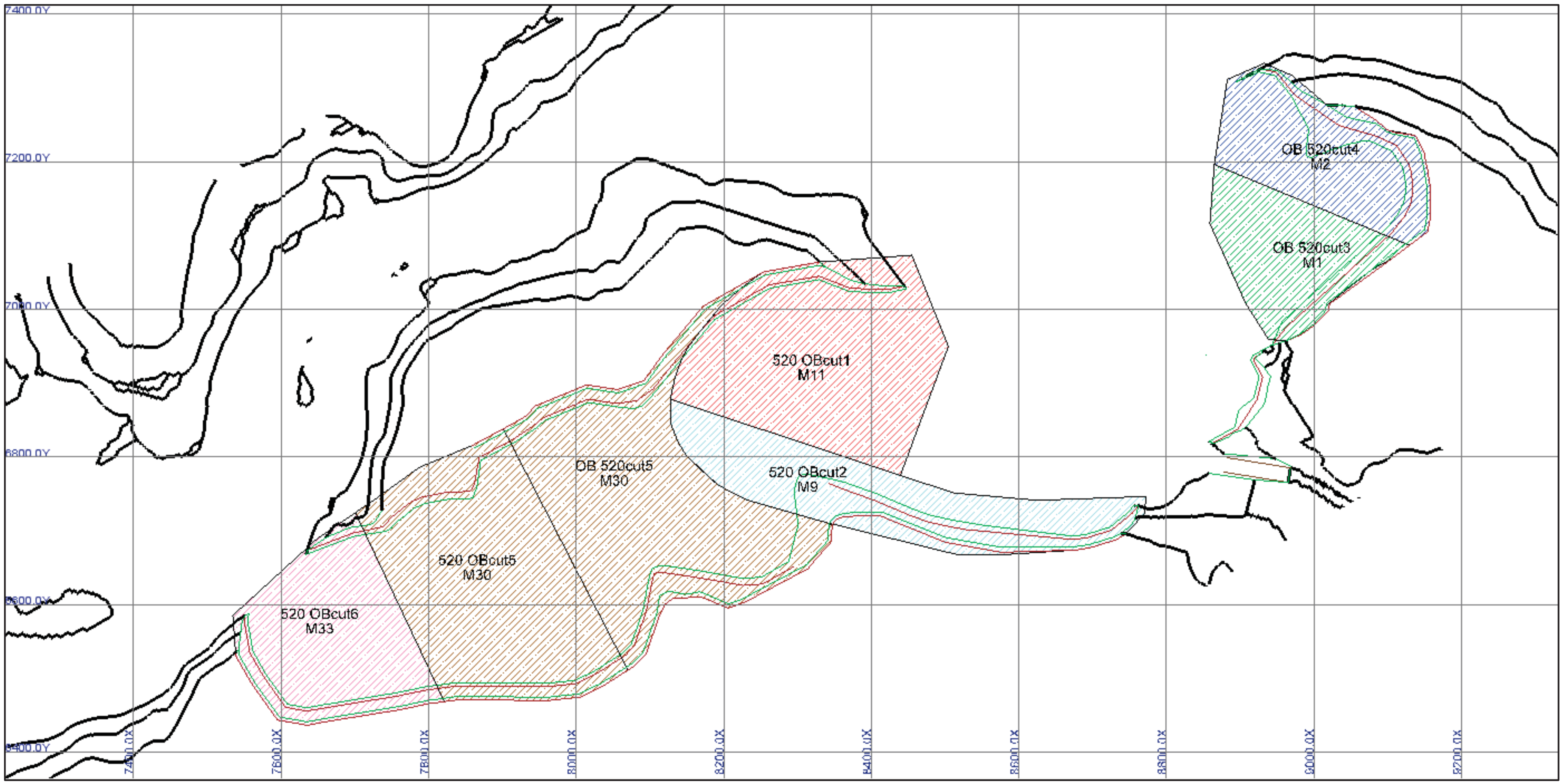

Figure 26 - 520 Bench Cut Showing Historic UG Workings (YR1-YR4)

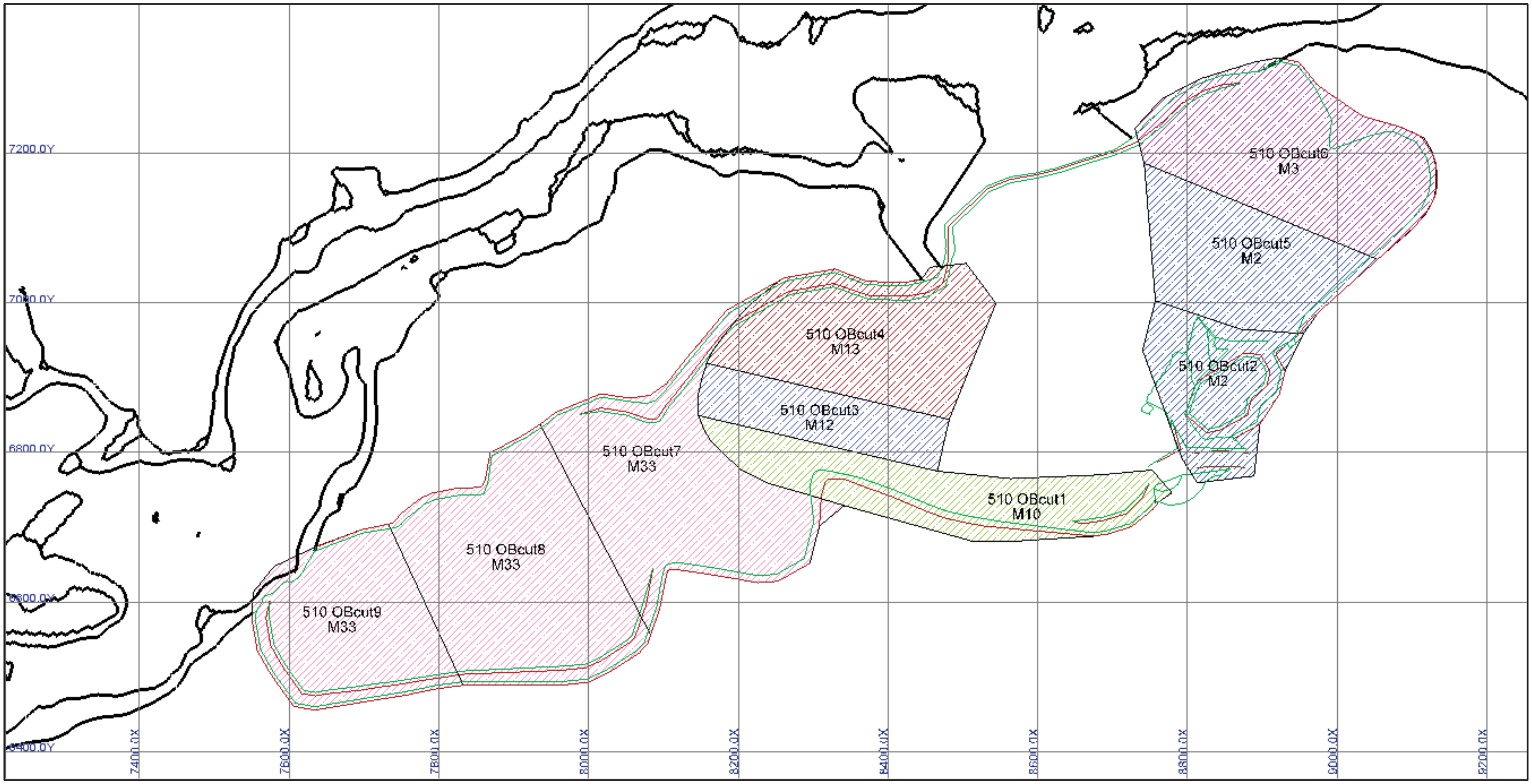

Figure 27 - 510 Bench Cut Showing Historic UG Workings (YR1-YR4)

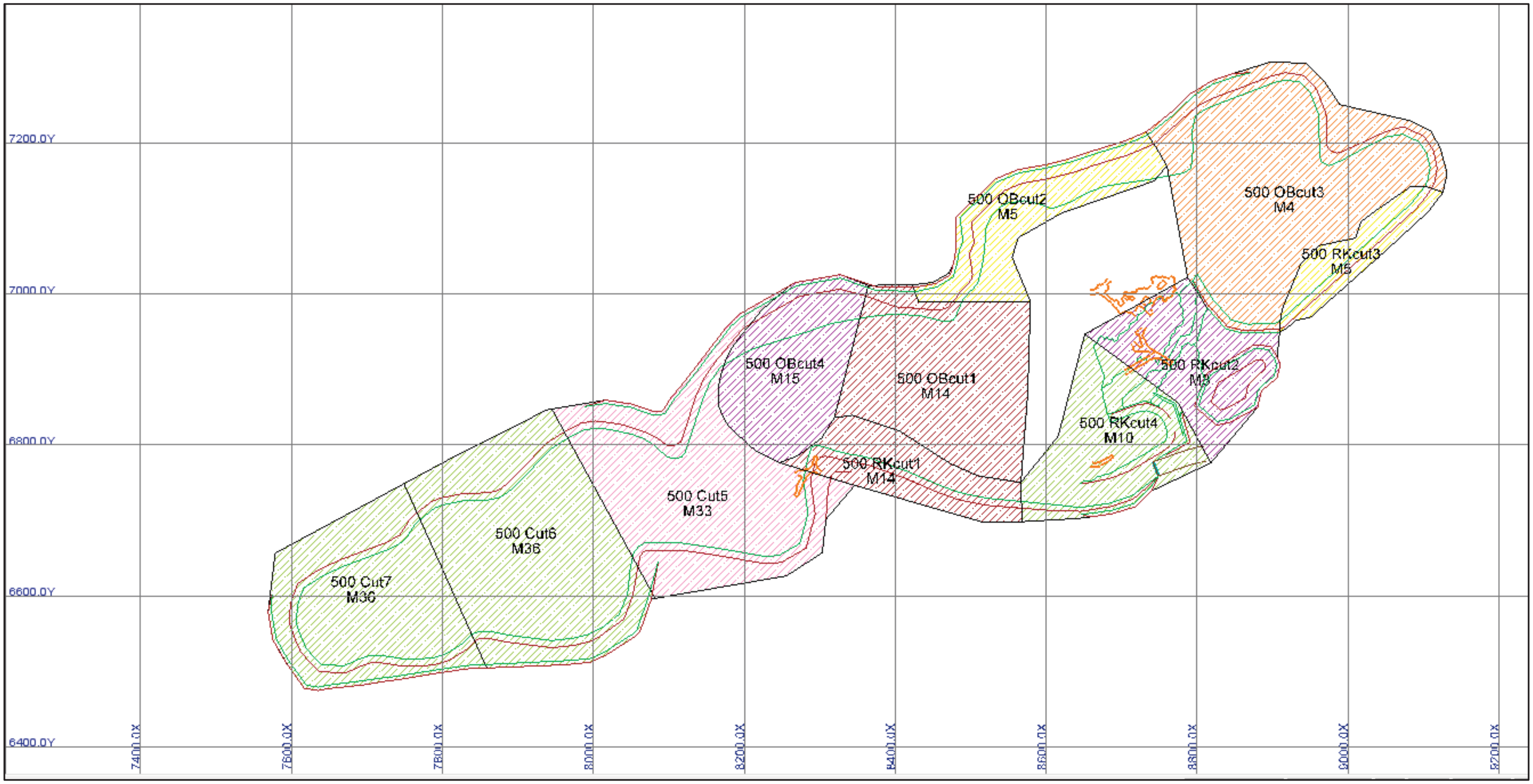

Figure 28 - 500 Bench Cut Showing Historic UG Workings (YR1-YR4)

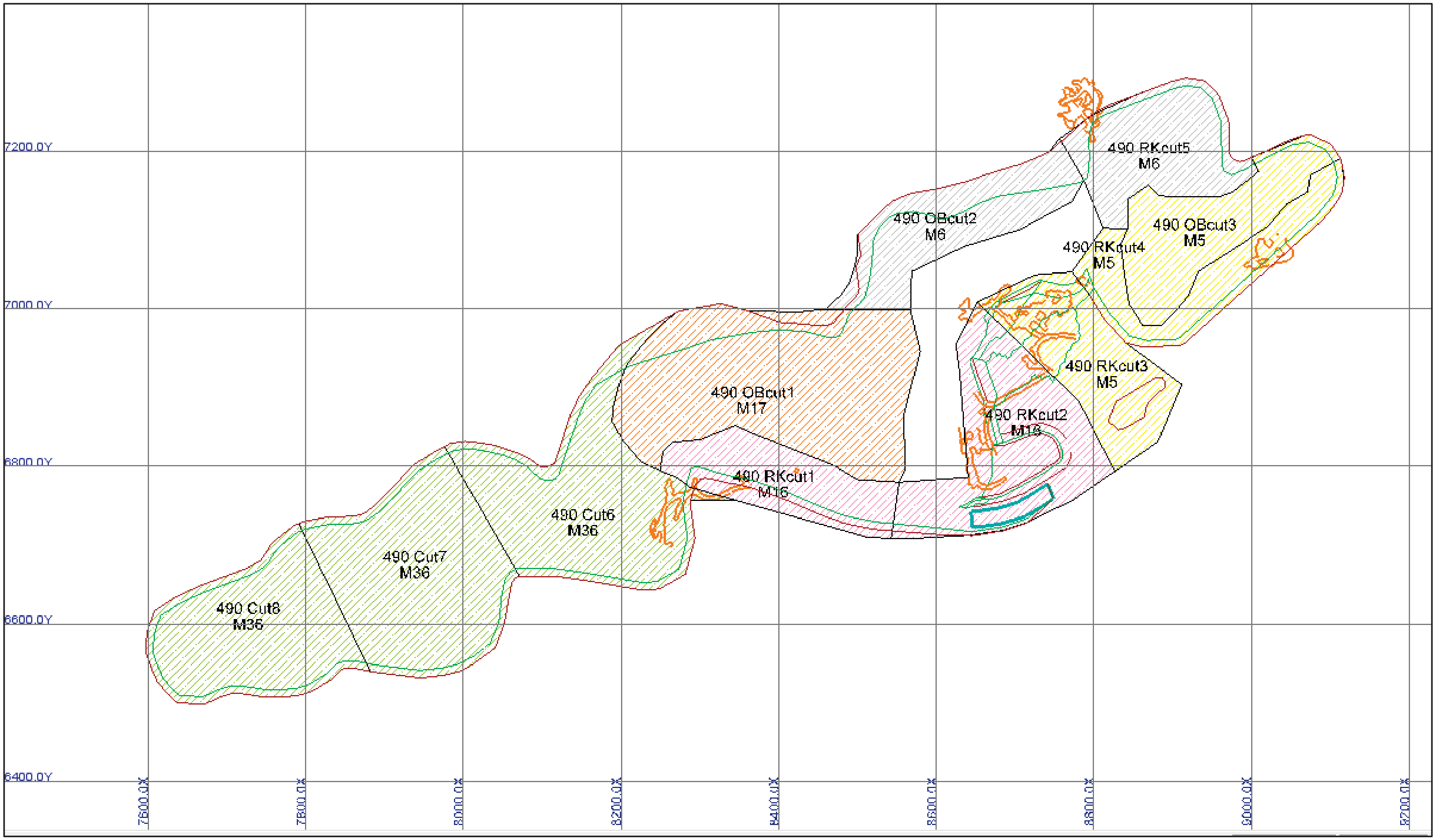

Figure 29 - 490 Bench Cut Showing Historic UG Workings (YR1-YR4)

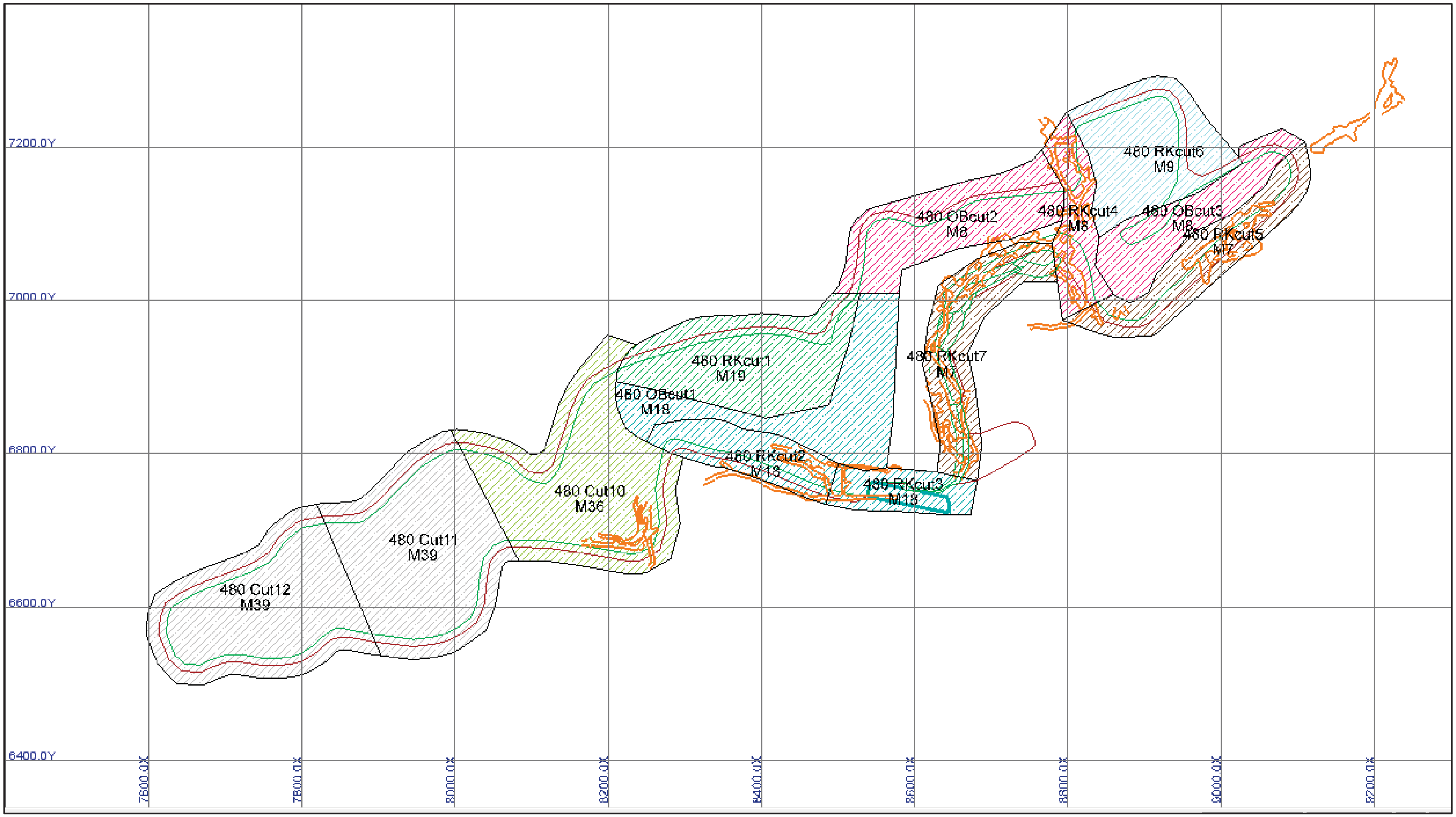

Figure 30 - 480 Bench Cut Showing Historic UG Workings (YR1-YR4)

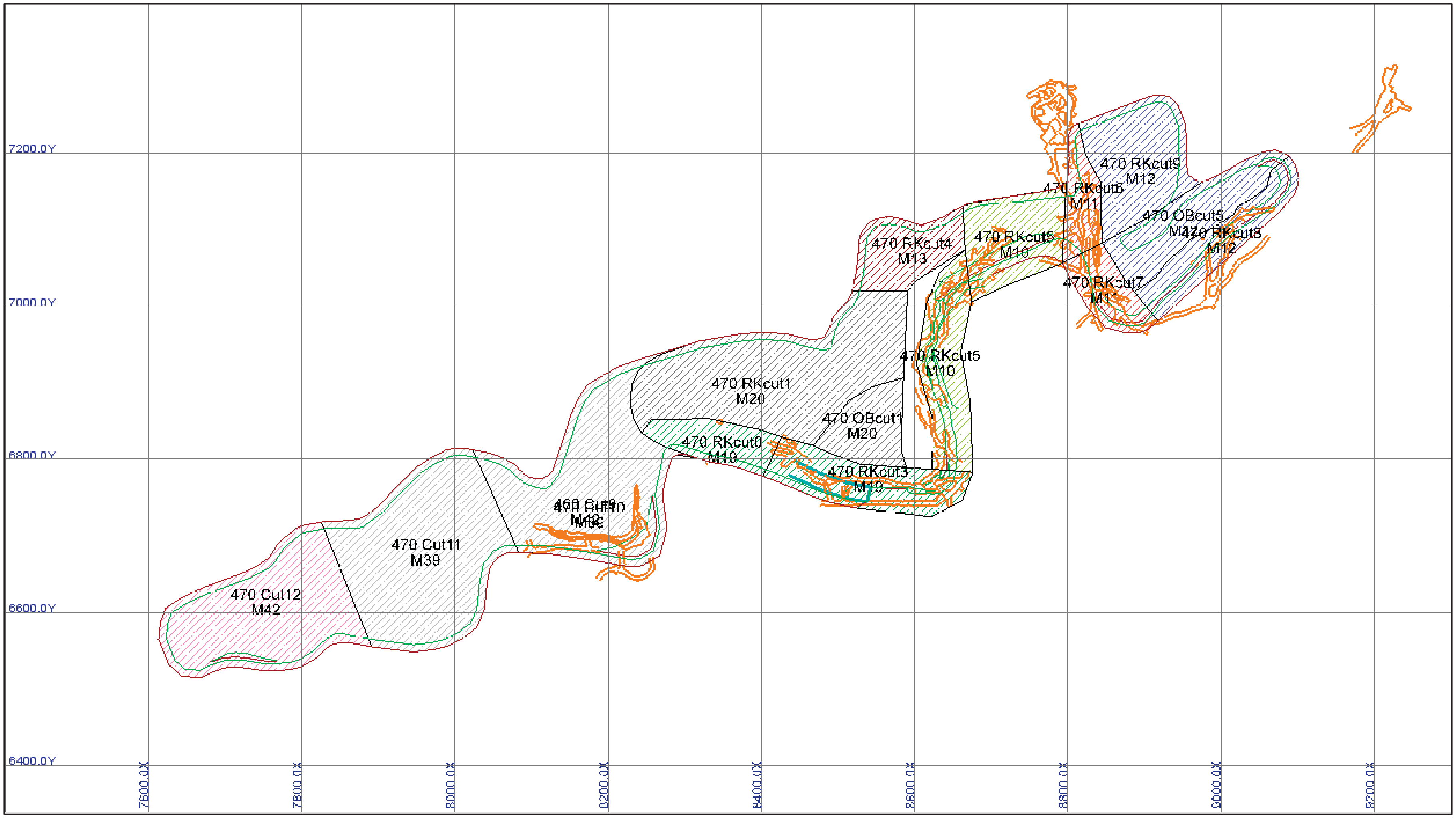

Figure 31 - 470 Bench Cut Showing Historic UG Workings (YR1-YR4)

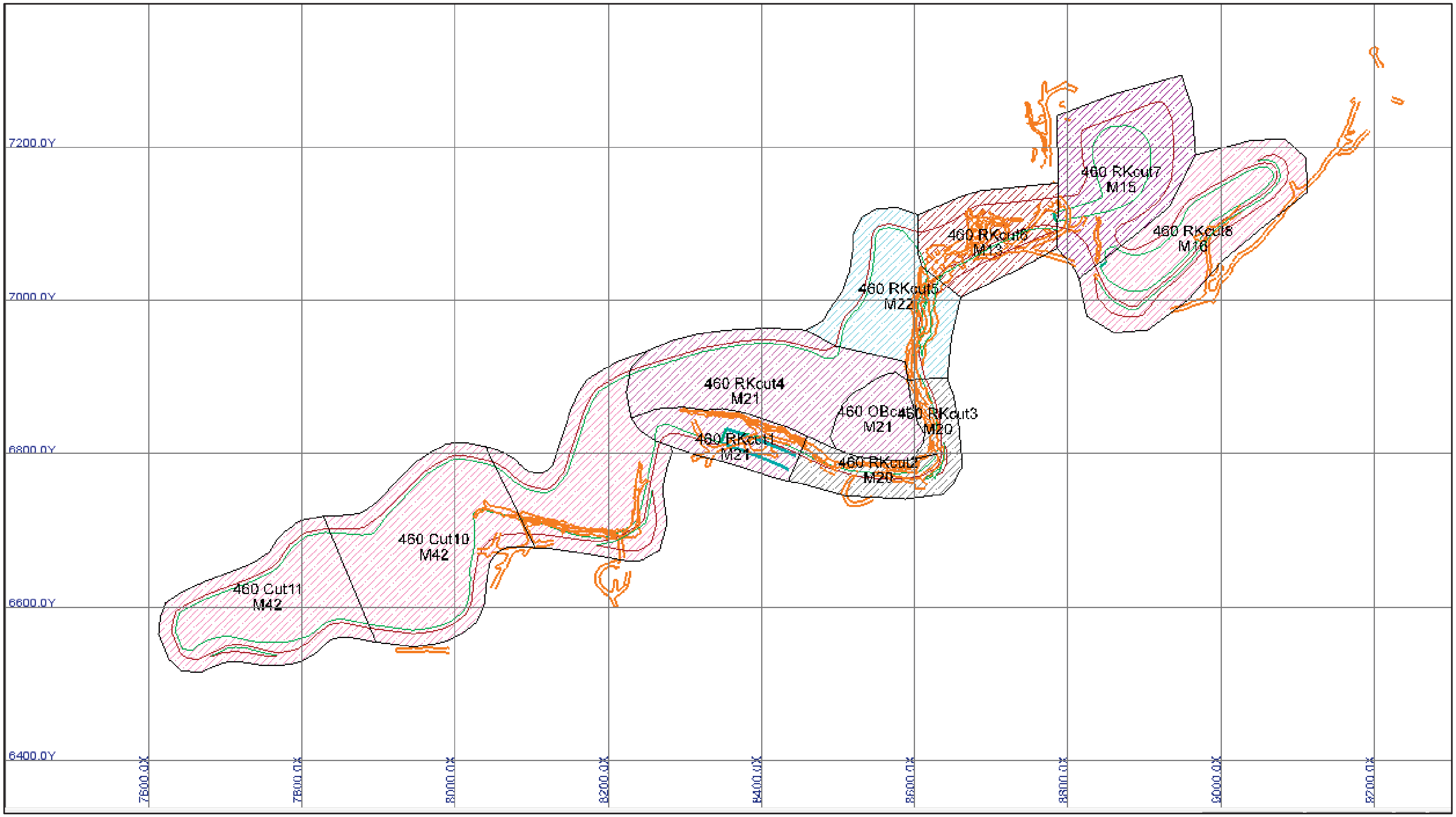

Figure 32 - 460 Bench Cut Showing Historic UG Workings (YR1-YR4)

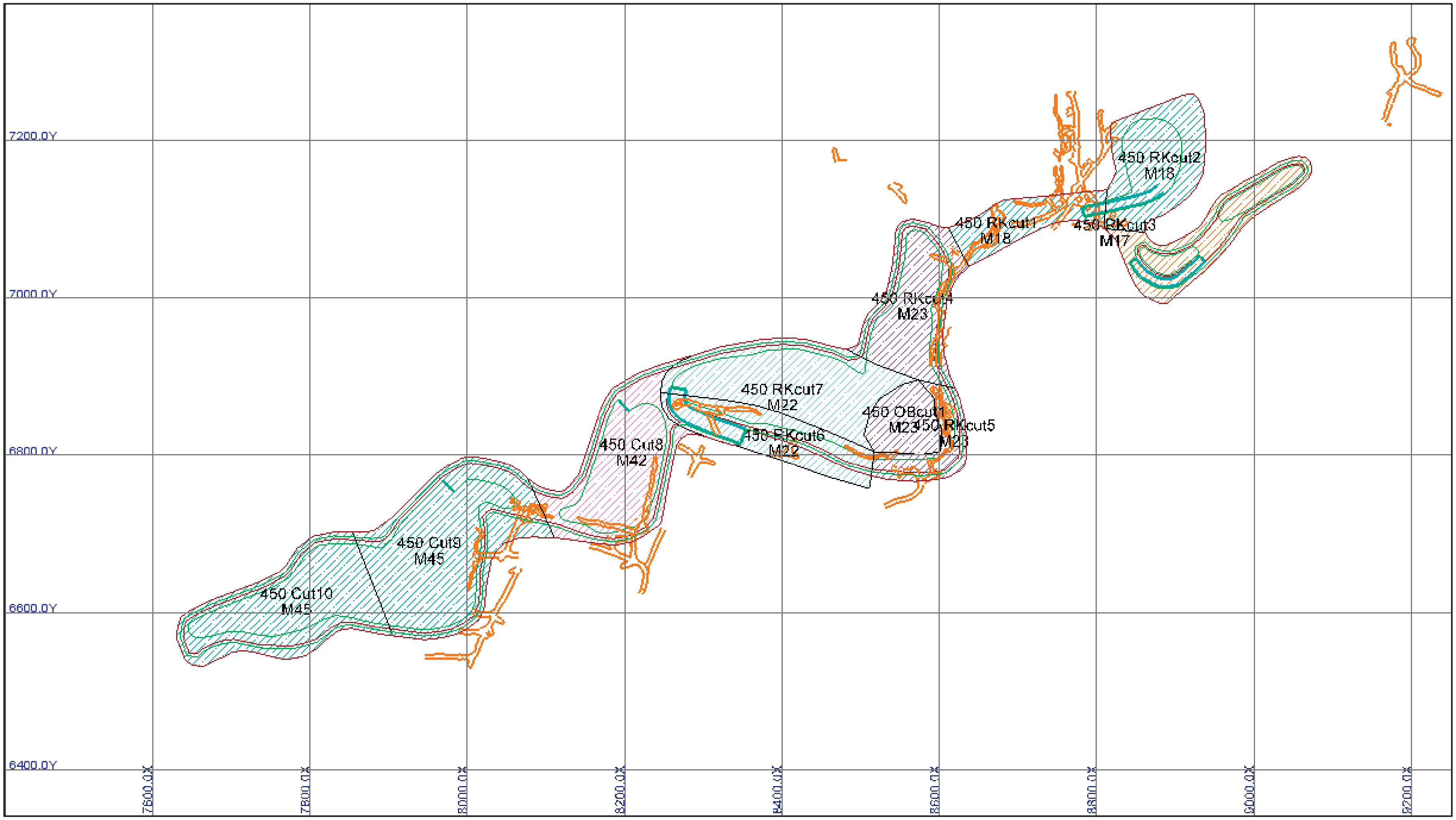

Figure 33 - 450 Bench Cut Showing Historic UG Workings (YR1-YR4)

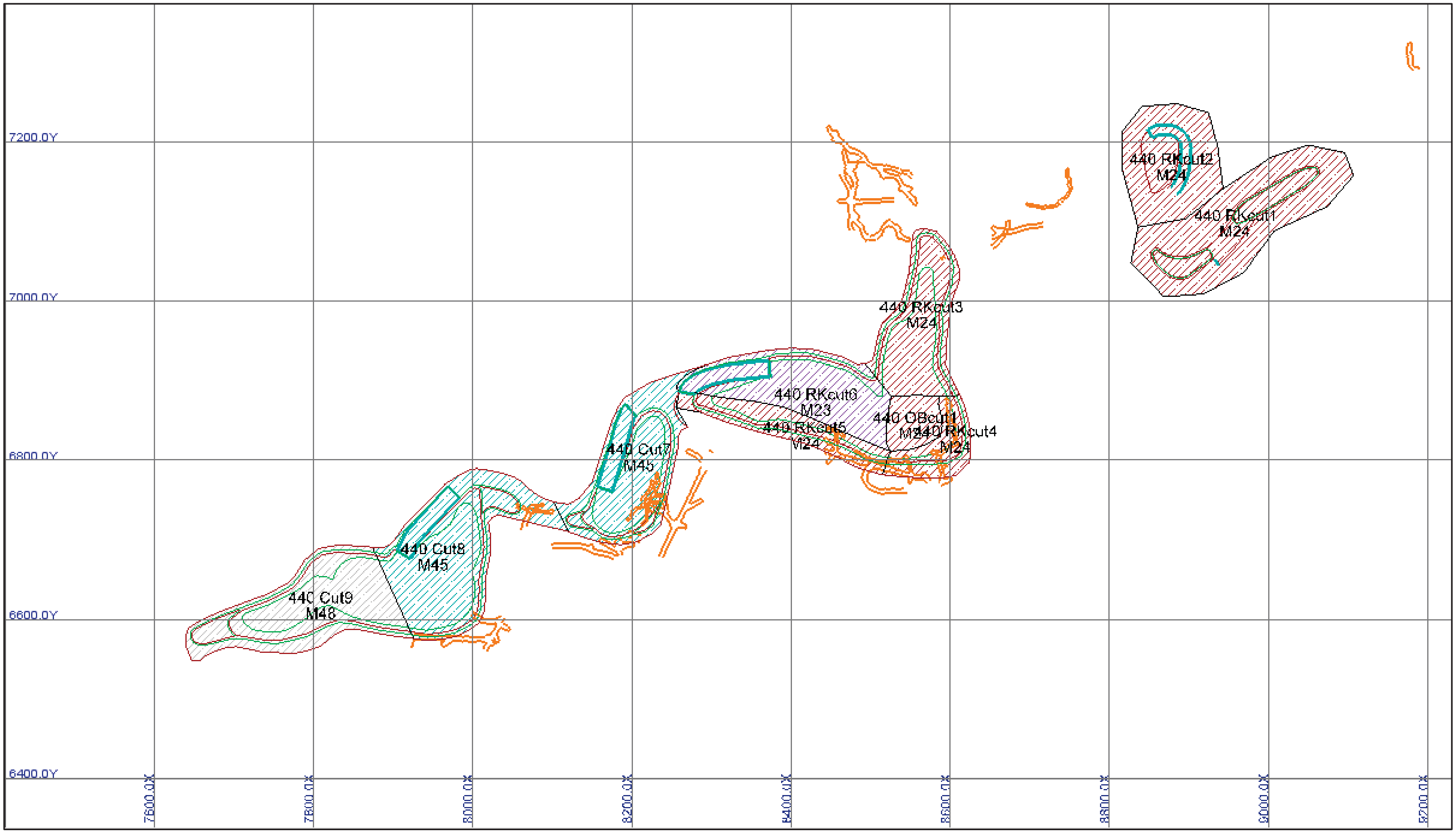

Figure 34 - 440 Bench Cut Showing Historic UG Workings (YR1-YR4)

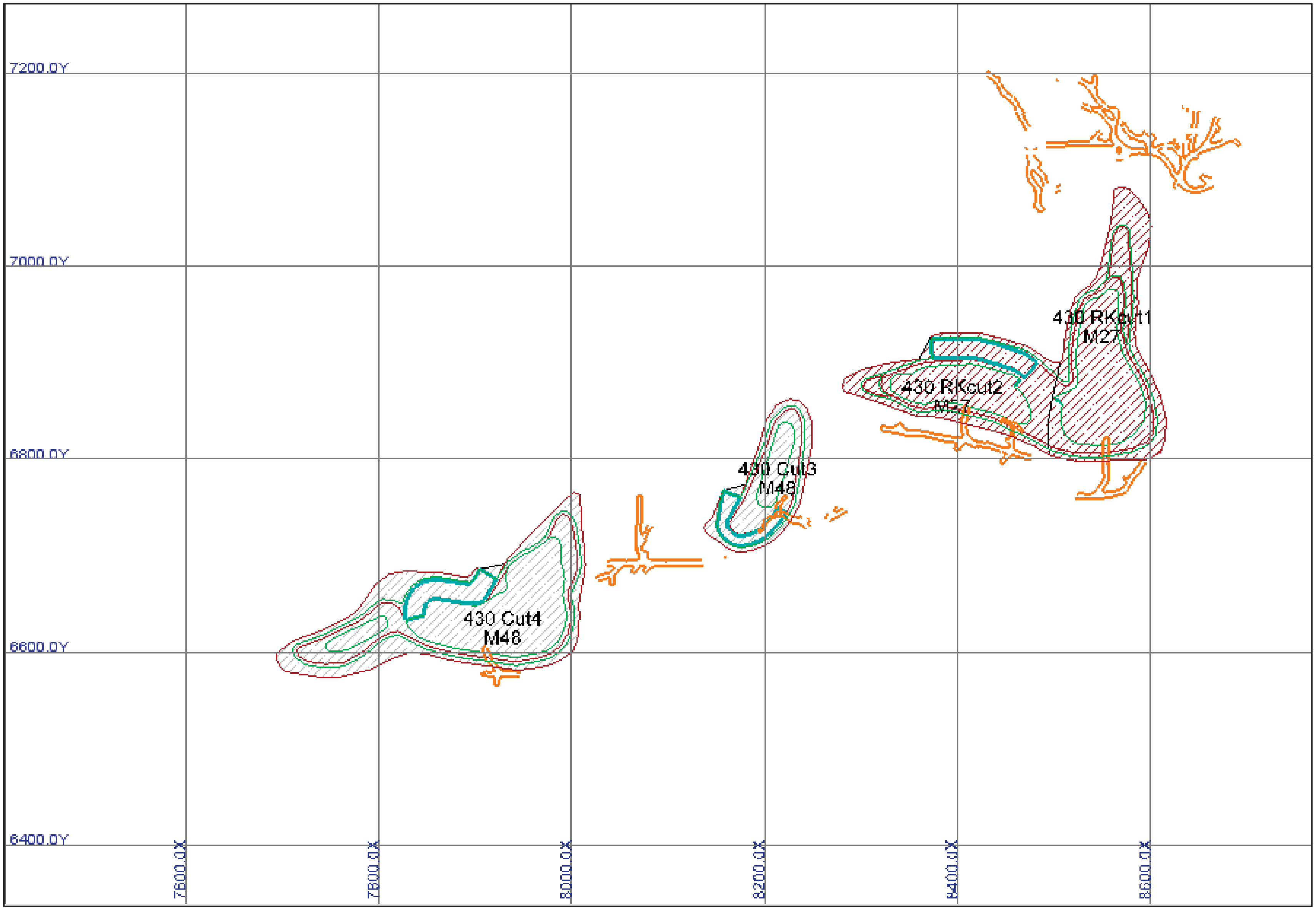

Figure 35 - 430 Bench Cut Showing Historic UG Workings (YR1-YR4)

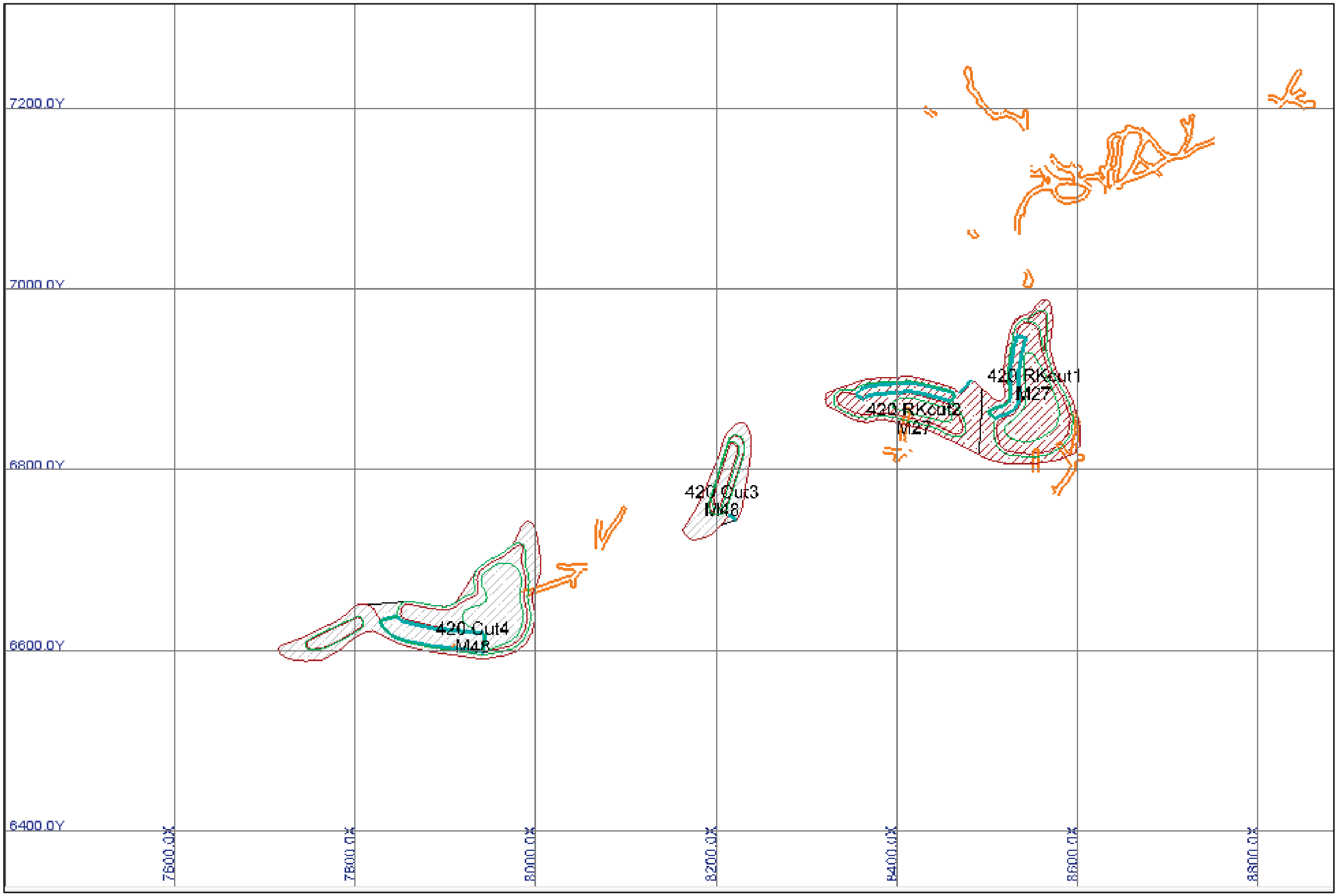

Figure 36 - 420 Bench Cut Showing Historic UG Workings (YR1-YR4)

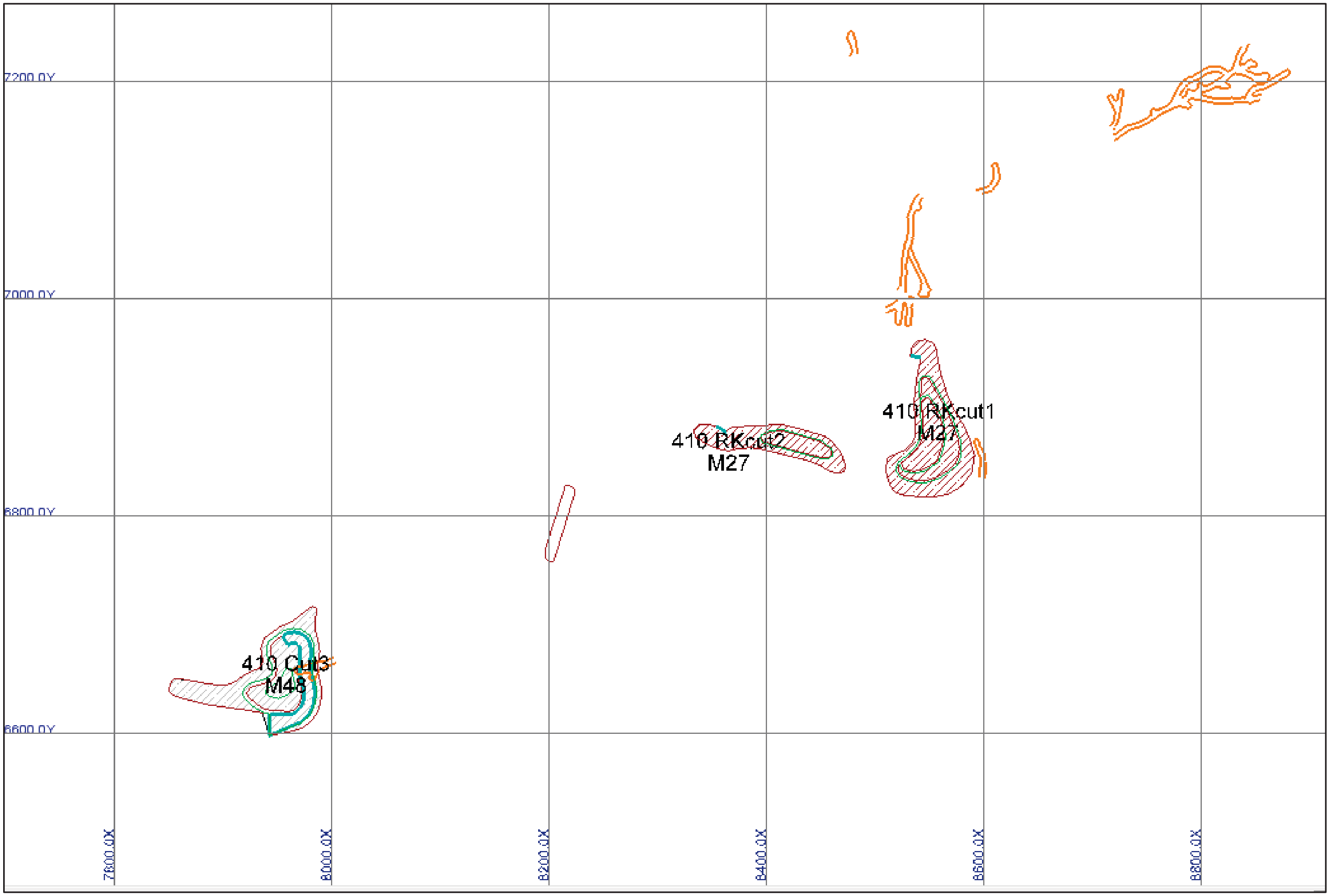

Figure 37 - 410 Bench Cut Showing Historic UG Workings (YR1-YR4)