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AVALON ADVANCED MATERIALS INC.

THE EAST KEMPTVILLE TIN PRODUCTION AND SITE REMEDIATION PROJECT PRELIMINARY ECONOMIC ASSESSMENT NOVA SCOTIA, CANADA

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OVERVIEW OF THE EAST KEMPTVILLE PROJECT

Micon International Limited (Micon) has been retained by Avalon Advanced Materials Inc. (Avalon) to prepare a Technical Report under Canadian National Instrument (NI) 43-101 which discloses the results of the preliminary economic assessment (PEA) for the East Kemptville Tin Project (East Kemptville Project), Yarmouth Co., Nova Scotia.

Avalon intends to recommence operations at the East Kemptville tin mine in Nova Scotia, Canada and in so doing will concurrently rehabilitate the mine site by remediating the existing environmental liability.

The re-development model, as presented in this PEA, is essentially an environmental remediation project that will be financed through the sale of conflict-free tin concentrates recovered in large part from previously-mined mineralized material on the site. From Day 1 of operations, it is Avalon's intent to continually reduce the long-term environmental liability and eventually result in the full rehabilitation of this brownfields site.

The PEA mine plan as developed by Micon is based on the updated mineral resource estimate disclosed in the Company's new release dated June 28, 2018. The redevelopment model primarily involves processing of the 5.87 million tonne (Mt) stockpile of previously-mined oxidized low-grade mineralization, supplemented by the selective mining of 9.2 Mt of near-surface fresh higher-grade tin mineralization from the Main and Baby Zone deposits.

The development model utilized by the PEA contemplates a production schedule averaging 1,300 tonnes per annum of a 55% tin concentrate for 19 years, with tin concentrates being sold and shipped for treatment in international markets. The PEA concludes that the small-scale re-development model is economically viable at current tin prices in the range of USD20,000 to USD22,000/t. Assuming an average go-forward tin price of USD21,038/tonne (as forecast by the World Bank Commodity Price outlook for 2020), and an exchange rate of CAD1.30/USD, the Project has an indicated pre-tax IRR of 15.0% and an NPV of CAD17.9 million at an 8% discount rate. This is after taking into account all costs associated with the proposed environmental rehabilitation process. The initial capital cost is estimated at CAD31.5 million and average annual revenues from sales are calculated as CAD17.75 million vs. annual production costs of CAD11.6 million.

The rehabilitation plan involves both the removal of acid generating surface stockpiles together with the concurrent sealing of the existing tailings storage facility (TMF) to prevent further acid generation. The freshly mined tin mineralization from the Main and Baby Zones will serve an important purpose in the site rehabilitation concept by allowing for the generation of clean tailings (by froth flotation) free of sulphide minerals. These clean tailings will be used to create the cover for the existing TMF which once rehabilitated can then be returned to beneficial use. The zinc/indium/copper/iron sulphide mineralization removed by the flotation will be appropriately disposed of underwater within one of the mined-out pits although recent rises in prices for some of these metals could result in the material becoming a source of additional revenue once up-graded.

Micon understands that Avalon anticipates that through the utilization of an existing TMF and site infrastructure as well as the focus on environmental remediation, the permitting and approvals process for the project will be much shorter than for a new, greenfield mine development. There is strong community support for the Project, as well as from local politicians, First Nations and environmental NGOs. Avalon is also in discussions with a number of local businesses towards collaboration on future opportunities during project development, throughout the operating life and for utilization of the site once rehabilitated.

While the results of the PEA indicate economic potential, there are a number of opportunities that Avalon is pursuing that could further improve Project economics. One of the most promising of these is the potential to upgrade the feed material to the processing plant through ore-sorting. Results from an initial evaluation were very encouraging and the results from a second evaluation are currently awaited. If these results reflect those of the earlier work, then further testwork or a piloting program is being considered.

Successful application of the ore-sorting process technology offers a number of benefits to the Project model. By rejecting non-mineralized waste rock ahead of the mill, the capacity of the processing plant can be reduced and both capital and operating costs lowered. It may also open the door for economic recovery of tin from other mineralized materials close-by that are presently considered too low in grade to justify processing. Alternatively, ore-sorting could provide the means to significantly increase annual tin production from the same sized processing facility over a shorter operating life.

It is Micon's understanding that prior to completing this PEA, the East Kemptville Project already attracted strong interest from a number of potential partners as well as others interested in securing off-take of the tin concentrates (which are in short supply from non-conflict sources). Avalon has already signed an indicative off-take agreement in the form of a non-binding Memorandum of Understanding (MOU) with a well-known, large tin smelting company for all of the forecast East Kemptville tin production.

1.0 SUMMARY

Micon International Limited (Micon) has been retained by Avalon Advanced Materials Inc. (Avalon) to prepare a Technical Report under Canadian National Instrument (NI) 43-101 which discloses the results of the preliminary economic assessment (PEA) for the East Kemptville Tin Project (East Kemptville Project), Yarmouth Co., Nova Scotia.

This PEA has been prepared by Micon under the terms of its agreement with Avalon. As discussed in the relevant sections of the report, Micon has prepared a mine plan and schedule, has reviewed the metallurgical testwork carried out on the property, the mineral processing flowsheet, has reviewed infrastructure requirements, prepared capital and operating cost estimates and an economic analysis of the project.

The PEA is based on the open pit mining and processing of mineral resources contained within two existing pits and an existing low-grade stockpile to produce a tin concentrate only. One important aspect of this relatively small-scale mining project is that it provides for a reduction in the long-term environmental liability and eventual full rehabilitation of the brownfield site.

The effective date of the mineral resource estimate on which this PEA is based, is 7 May, 2018 (see Avalon press release dated 28 June, 2018). This resource estimate was prepared by Avalon and is considered not to be materially different from the previous mineral resource by Hains Engineering Company Limited (Hains Engineering), which is described in an Avalon press release dated 31 October, 2014. Hains Engineering is independent of Avalon.

The Qualified Persons for this Technical Report are the following:

- Dayan Anderson, M.S., MMSA (Micon).
- Christopher Jacobs, CEng., MIMMM (Micon).
- Richard Gowans, P.Eng. (Micon).
- Jane Spooner, P.Geo. (Micon).
- William Mercer P. Geo. (Avalon).
- Donald H. Hains, P.Geo. (Hains Engineering)
- Reid Smith P.Geo. (Stantec)

1.1 EAST KEMPTVILLE PROPERTY

The East Kemptville tin-indium project is located on NTS map sheet 21A/04A and /05B in Yarmouth County, southwestern Nova Scotia. The property is located approximately 180 km southwest of Halifax, the provincial capital, and 55 km northeast of the town of Yarmouth. The site accessed from Yarmouth via Nova Scotia paved Highways 340 to Carleton and then 203 to the site. Yarmouth lies on Highways 103 and 101, approximately 300 km by road from Halifax.

Avalon holds a 100% interest in the property via Special Licence 50462. The area covered by Special Licence 50462 includes the Closure Area represented by the former East Kemptville Tin Mine property, which is currently under the management of Rio Algom Ltd (RAL), the surface rights holder. The Special Licence 50462 issued to Avalon on 24 April, 2015 by the Government of Nova Scotia, is for a term of three years, renewable twice for one year. While Avalon's Special Licence 50462 is active, the Mineral Resources Act provides protection against competing applications to parties with active applications under consideration. As of the date of this report, the Special Licence 50462 has been renewed by the Government of Nova Scotia to 2 February 2019.

1.2 History

Significant, greisen-style tin mineralization was discovered in granitic outcrop in the East Kemptville area, in 1978, by Shell Canada Resources Limited (Shell). Shell initially drilled a total of 136 diamond drill holes for a total of 12,450 m during 1979 and 1980, followed by a further 23 diamond drill holes totalling 1,840 m in the centre of the deposit to test for grade continuity between existing holes. Subsequently, an additional four diamond drill holes totalling 490 m were drilled as part of an underground exploration and bulk sample program conducted between September, 1981 and February, 1981.

The south-central part of the deposit was also tested by 975 m of underground drifting during the period from September, 1980 to February, 1981. The ramp access tested an area of approximately 500 m by 350 m to a vertical depth of 50 m. A total of 31,600 t of material was extracted as a bulk sample and four underground diamond drill holes totalling 490 m were drilled for comparative purposes. (RAL, 1983)

In 1982, the East Kemptville Deposit and surrounding claims were purchased from Shell by Riocanex, the Canadian exploration arm of RAL. During 1982 to 1983, RAL conducted a detailed due diligence of Shell's work and drilled a total of 15 drill holes totalling 1,305 m during 1983 in preparation for a feasibility study and production decision also completed in 1983.

The open-pit operation at East Kemptville commenced in the fall of 1985 with a reported planned 17 years of production at rates of 9,000 t/d of plant feed material and 5,000 t/d of waste. This operation produced high (50% Sn) and low (21.4% Sn) grade tin concentrates a copper concentrate (25% Cu) and a zinc concentrate (50% Zn). Shortly after commencing production, the operation ran into serious problems related to the recovery of tin by gravity methods. A dramatic price decline of approximately 50% for tin on world markets in the fall of 1985 put added pressure on the operation. Continued poor tin prices resulted in cessation of operations in early 1992.

1.3 Geology and Mineralization

The East Kemptville Project is located within the Cambro-Ordovician aged, Meguma Terrane of mainland Nova Scotia. The East Kemptville deposit is a greisen hosted Sn-Cu-Zn-

Ag-In deposit with the alteration and mineralization mostly affecting the East Kemptville leucogranite (EKL).

Tin and base metal (Zn-Cu-Ag-W) mineralization within the deposit is primarily fine to medium-grained and is associated with northeast-trending, sub-vertical and zoned, quartz-topaz, sulphide-bearing greisens, veins, and stockworks that occur primarily in the sericite-silica-topaz altered portions of the EKL near where the East Kemptville Shear Zone (EKSZ) meet the roof zone in contact with surrounding metasediments.

The overall gross dimensions of the original potential economic mineralization at the Main and Baby Zones based on a cut-off grade of approximately 0.05% Sn are in the order of 1,500 m long, 350 m wide and 75 m to 150 m deep. Most of this volume is represented by the larger, Main Zone. The smaller and discrete Baby Zone occurs a few hundred metres southwest of the Main Zone within what is believed to be a structurally controlled, satellite intrusion. Mineralization between the Main Zone and the Baby Pit is referred to as the Southwestern Extension of the Main Zone and is not exposed at surface but intersected in drilling.

Cassiterite accounts for most (>90%) of the tin mineralization with stannite accounting for the remainder. Zinc is primarily found as sphalerite and indium is associated with the sphalerite. Copper is primarily present as chalcopyrite and other copper sulphide minerals.

1.4 EXPLORATION

Prior to the 2014 and 2015 drill programs, exploration by Avalon has been limited to regional reconnaissance geochemical sampling and limited diamond drilling on the exploration licences outside of the Special Licence area.

1.4.1 Avalon 2014 Drilling Program

Avalon completed an in-fill/twin hole program consisting of seven HQ diamond drill holes totalling 986 m in 2014. The purpose of Avalon’s 2104 drill program was to investigate mineralization between the Main Zone pit and Baby Pit, referred to as the Southwestern Extension of the Main Zone, and at depth, and to twin some selected historic holes as part of a due diligence program to validate the historical drill results.

In general, the geology and polymetallic Sn + Zn + Cu zones encountered in the drilling are considered to be typical of historic drill results reported by Shell and RAL in the Baby and Southwest Extension Zone Areas. Drilling was successful in confirming the known geology and the mineralization associated with the Southwest Extension of the Main Zone

1.4.2 Avalon 2015 Drilling Program

In 2015, Avalon completed the drilling of twenty-two HQ diamond drill holes totalling 4,514 m. The objectives of this program were to further definition of mineral resources, obtain additional geotechnical information for mine planning and geochemical information for waste rock handling planning, and obtain a bulk sample for potential pilot scale metallurgical testing.

The drill hole sample preparation and assays conducted under best practice QA/QC procedures with insertion of blanks and standards, as well as duplicate coarse sample analyses at a secondary laboratory and core duplicates.

1.5 SAMPLE PREPARATION, ANALYSES AND SECURITY

1.5.1 Avalon 2014 Drilling Program

Core was placed in numbered and marked core boxes at the drill site and a quick log prepared. Avalon personnel transferred boxes to the core logging area where drill core was logged in detail and marked for sampling and all core photographed prior to sampling.

Sampling was typically undertaken on 1.5 m intervals within mineralized sections. Core was split using a manual core splitter, with the remaining ½-core reassembled in order in the core box. Sample material was placed in plastic sample bags with sample number marked on the outside of the bag and a sample tag stapled to the inside fold of the bag. A duplicate tag was placed in the core box. Duplicate samples were obtained from drill core by splitting core in half, with one half noted as the main sample and the other half noted as the duplicate in the sample log. Standards and blanks were inserted in the sample list on a pre-determined basis.

Bagged samples were placed in 20-L plastic pails. The pails were sealed with secure lids and taped closed and the sample numbers noted on the outside of the pail. Once a sufficient number of samples had been prepared, samples were shipped by courier to ALS Canada Ltd. (ALS) in Sudbury for initial sample preparation. After initial sample preparation, ALS shipped the samples to its Vancouver facility for assaying. Check sample splits were shipped by ALS to SGS Canada in Lakefield, Ontario, and to Activation Laboratories Ltd. (Actlabs) in Ancaster, Ontario.

A total of 404 samples (excluding the 57 duplicates, standards and blank samples) were submitted to ALS for multi-element analyses. Sixteen blanks, 15 standards and 13 field duplicates were inserted into the three sample shipments to monitor contamination, accuracy and precision.

1.5.2 Avalon 2015 Drilling Program

For the 2015 drill program, the sample treatment at the core logging facility was similar to 2014, with the exception that the samples were shipped to Actlabs' sample preparation facility in either New Brunswick or Ontario, with the New Brunswick facility utilized except in some cases where a backlog had built up in New Brunswick. In the latter case, the samples were shipped direct to the Actlabs laboratory in Ancaster, Ontario.

For the 2014 program, only a limited amount of sampling was undertaken in non-mineralized sections (limestone and greywacke). However, in 2015, in light of the occurrence of mineralization to the boundary of sampling, prior to the start of the 2015 drill program, additional sections of unsampled 2014 core were split and assayed and, in some cases, contained significant mineralization that was contiguous with existing known mineralization.

The initial sample processing and analysis was completed by Actlabs (Ancaster, Ontario) and the check samples sent to ALS (Vancouver, BC) for analysis.

1.5.3 Low Grade Stockpile Surface Sampling Program

In order to verify the metal grade of the low-grade stockpile, a surface sampling program was completed in 2015. A program was completed with two samplers to reduce sample bias, each independently taking a sample at points at 50 m intervals across the length and width of the low-grade stockpile, plus samples around one side of the bottom of the pile. The two samples from the two individuals from each site were kept separate for analysis in order to investigate any sampling bias on the part of one or other sampler. A total of approximately 270 kg was collected with each sample being about 5 kg.

Samples collected from each site were shipped to Actlabs for analysis. Comparing these analyses with the RAL Closure Plan (RAL, 1993) showed that the Avalon estimates for Sn and Zn grades are within 11% of the surface samples quoted by RAL.

1.6 DATA VERIFICATION

Data verification for the 2014 drill program and the resource database included the following:

1. Comparison of 2014 drill core sample numbers against assay sample shipment lists and sample receipt list.
2. Survey of drill collar coordinates by a qualified Nova Scotia land surveyor.
3. Site visit and inspection of 2014 drilling procedures, core logging, and sampling by Hains Engineering.
4. Collection of due diligence ¼-core samples and independent assaying of samples by Hains Engineering.
5. Comparison of 2014 drill core assays against assay certificates.
6. Comparison of historic drill logs and assay certificates against the historic Excel database used in the resource estimate.
7. Verification of historic Rio Algom Limited (RAL) QA/QC data.
8. Inspection of selected historic drill core stored at the NSDNR Core Library in Stellarton, Nova Scotia and verification of descriptions in historic drill logs by Hains Engineering.

9. Collection of due diligence ¼-core samples and independent assaying of samples from selected drill core intervals of historic drill core stored at Stellarton by Hains Engineering.

In the opinion of author, the 2014, sampling and assay data and the historic drill hole and assay data, as represented in the resource database, are reliable and can be used in resource estimation.

QA/QC measures employed for the 2014 drill program included the following:

1. Insertion of standards and blanks in the main sample batches.
2. Assays of coarse duplicates to check sample preparation procedures and laboratory precision.
3. Assays of pulp duplicates to laboratory analytical precision.
4. Coarse check samples assayed at two separate laboratories to check sample preparation procedures and analytical bias.
5. Insertion of certified standard reference materials in check sample assay batches.
6. Internal laboratory QA/QC protocols incorporating the use of certified standards and blanks and duplicate and repeat assays.

The review of the QA/QC data indicates no significant issues with respect to sample preparation, assaying and laboratory precision.

Similar QA/QC protocols were followed in the 2015 drill program as previously used in 2014. The results on the standards and duplicates suggest that Actlabs may have a slight negative bias in analyses in tin. As all biases present are indicated at levels below 10% and in most cases less than 5%, the analytical data is considered acceptable for resource estimation.

Data verification of the historic sampling and assay data consisted of checking the reported assay values contained in the QA/QC appendix of the RAL feasibility study against the current assay data base and the available drill logs. The current assay data base is a compilation undertaken by Avalon of all available assay certificates, drill logs and survey data. The Shell and RAL drill core assays are considered as acceptable for resource estimation purposes.

1.7 MINERAL PROCESSING AND METALLURGICAL TESTING

Avalon has conducted a number of testwork programs on samples representing the East Kemptville deposit. Work began with SGS UK in Cornwall, UK, to develop a comprehensive flowsheet to produce tin, copper, and zinc/indium concentrates using mineralized samples from the Baby Zone deposit. This set the baseline for a subsequent test campaign, in 2016, at Met-Solve, who investigated recovering tin (only) from the existing low-grade stockpile.

Testwork was undertaken by SGS UK using a 290 kg blended composite from 1,140 kg of material comprising 394 split drill core from the Baby Zone. The testwork completed included heavy liquid sink/float tests, Bond rod and ball mill grindability tests, gravity separation tests and flotation tests.

SGS UK was able to develop a flowsheet for the East Kemptville deposit to produce copper, zinc and tin concentrates. Copper recovery was estimated at 86.4% into a 20.7% grade copper concentrate, zinc recovery was estimated at 84.5% into a 51.4% grade zinc concentrate and tin recovery into a 50.5% Sn concentrate was estimated at 76.8%.

Using the SGS UK test results as a basis, Avalon contracted Met-Solve in Langley, BC (Met-Solve), in 2016, to undertake further flowsheet development testwork to recover tin from East Kemptville mineralization. The testwork program was divided into three phases:

- Phase I: Use of falcon gravity concentrators at 3 different grind sizes (200, 150 and 100 µm) to determine the sample's response to gravity concentration for the recovery of tin.
- Phase II: Grind material to 200 µm for the gravity rougher stage, followed by a regrind to 100 µm for gravity scavenging. Gravity tailings were then floated to attempt to recover additional tin.
- Phase III: Locked Cycle tests of the best flowsheet configuration previously identified in Phases I and II.

The sample provided by Avalon for the Met-Solve testwork comprised approximately 178 kg of crushed samples from the East Kemptville low-grade stockpile.

The testwork resulted in the development of a flowsheet capable of producing a tin concentrate containing up to 55% Sn with a tin recovery of approximately 60%. The material will be milled to P_{80} ±80 microns before being put through a series of centrifugal gravity concentrators. The gravity concentrates will feed a magnetic separation circuit followed by a simple sulphide flotation circuit to remove sulphides. The non-sulphide flotation tailings will be cleaned using final shaking table gravity circuit, the concentrates from which will be collected and dewatered before being shipped to potential customers.

Testwork also showed that tailings from the gravity circuit can be treated through a bulk sulphide flotation process to reduce contained sulphur to approximately 0.05% S, making it a suitable material for capping of the tailings dam

1.8 MINERAL RESOURCE ESTIMATES

An updated mineral resource estimate for the East Kemptville project was completed on 7 May, 2018 (see Avalon news release dated 28 June, 2018). The mineral resource estimate is based on a block model prepared by Avalon and is summarized in Table 1.1. The deposit was subdivided into the Main Zone and the Baby Zone, which were interpolated separately. The in situ unmined tin resources were estimated using historic drill holes, data from drill holes completed by Avalon in 2014 and 2015, and a post-mining topographic model. A tin cut-off grade of 0.10% was considered as reasonable based on current mine plans and historic cut-off grade used at the East Kemptville mine.

Table 1.1
Updated Mineral Resource Estimate for the Main and Baby Zones

Classification	Cut-off grade Sn (%)	Main Zone NE		Baby Zone		Total	
		Tonnes (Mt)	Sn (%)	Tonnes (Mt)	Sn (%)	Tonnes (Mt)	Sn (%)
Measured	0.08	0.40	0.173	0.22	0.241	0.61	0.197
	0.10	**0.38**	**0.177**	**0.20**	**0.251**	**0.58**	**0.203**
	0.12	0.32	0.188	0.19	0.259	0.51	0.214
Indicated	0.08	27.89	0.133	1.72	0.194	29.61	0.137
	0.10	**20.91**	**0.148**	**1.48**	**0.211**	**22.39**	**0.152**
	0.12	14.84	0.163	1.27	0.228	16.11	0.168
Measured + Indicated	0.08	28.28	0.134	1.93	0.199	30.22	0.138
	0.10	**21.29**	**0.148**	**1.68**	**0.216**	**22.97**	**0.153**
	0.12	15.16	0.164	1.46	0.232	16.62	0.170
Inferred	0.08	18.54	0.125	0.90	0.153	19.43	0.126
	0.10	**13.56**	**0.137**	**0.69**	**0.172**	**14.25**	**0.139**
	0.12	8.11	0.156	0.51	0.193	8.62	0.158

Notes:
1. CIM Definition Standards for Mineral Resources, 2014, were followed.
2. The Qualified Person for this Mineral Resource estimate is William Mercer, Ph.D., P. Geo. (Nova Scotia). The mineral resources are current as of May 7, 2018.
3. The mineral resource estimate is based on 194 drill holes totalling 21,456 m drilled between 1979 and 1991 by previous operators and 23 holes totalling 4190 m drilled by Avalon in 2014 and 2015.
4. Drill data were organized in Maxwell DataShed and for estimation purposes were transferred to the Geovia GEMS 6.8.1 software, wherein the block model was developed.
5. Resources were estimated by interpolating composites within block models of 24 m by 24 m by 12 m blocks in the Main Zone and 6 m by 6 m by 6 m in the Baby Zone. Interpolation used the Ordinary Kriging method.
6. In the Main Zone, Measured material was defined as blocks interpolated with a search ellipse with radii of 40x20x15 m using 18-36 samples, corresponding to 3-6 drill holes, indicated material with a 120x40x18 m search ellipse and the same number of samples, and inferred material with a 315x85x18 m search ellipse using 12-24 samples corresponding to 2-4 drill holes. In the Baby Zone, Measured material was defined as blocks interpolated with a search ellipse with radii of 30x20x8 m using 6-12 samples, corresponding to 3-6 drill holes, indicated material with a 48x33x12 m search ellipse and the same number of samples, and inferred material with a 95x65x24 m search ellipse using 4-8 samples corresponding to 2-4 drill holes (see Section 1.12 Resource Classification).
7. Prior to compositing, the assays were capped at 1% Sn, which corresponds to the 99th percentile of the tin assay data, reducing the length-weighted mean of the tin assays by 9.4%.
8. Mean density values of available data of 2.728 t/m^3 and 2.784 t/m^3 were used for the Main and Baby Zones, respectively.
9. The resource estimate has been constrained using the Whittle pit described previously (Avalon News Release 15-02, February 25, 2015).

10. Several possible cut-off grades are reported in this resource estimate. Based on past mining practice at East Kemptville, a cut-off grade of 0.1% Sn is reasonable and preliminary cost and revenue values at the time of estimation also suggest this is reasonable.
11. Mineral resources do not have demonstrated economic viability and their value may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other issues.

The Qualified Person (QP) for the Baby and Main Zone mineral resources reported in the PEA is William Mercer, P. Geo. who is not independent from Avalon. However, these current resource estimates have not changed significantly since the previous versions that were prepared independently by Hains Engineering with its principal, Donald H. Hains (P.Geo), serving as the independent QP for the purpose of NI 43-101 (News Release 14-13, October 31, 2014).

There has been no change of the mineral resource estimate for the low-grade stockpile since the previous estimate by Hains Engineering with an effective date of 16 November, 2015 (see Table 1.2).

Table 1.2
Low Grade Stockpile Estimated Inferred Mineral Resource

Category	Tonnes (Mt)	Grade (%)		
		Sn	Zn	Cu
Inferred	5.87	0.112	0.100	0.61

Notes:
1. This estimate is as of 16 November 2015.
2. CIM Definition Standards 2014 were followed for mineral resources.
3. The independent Qualified Person for this Mineral Resource estimate is Donald Hains, P.Geo., of Hains Engineering.
4. Resources were estimated by examination of historical RAL data and Avalon's 2015 sampling of the Low-Grade Stockpile.
5. Mineral resources do not have demonstrated economic viability and their value may be materially affected by environmental, permitting, legal, title, socio-political, marketing or other issues.

1.9 MINERAL RESERVE ESTIMATES

No mineral reserves have been estimated for the East Kemptville tin project.

1.10 MINING METHODS

Avalon plans to engage a locally (Eastern Canada) based mining contractor to mine material from the Rio Algom Ltd. (RAL) legacy stockpile, the Baby Zone pit and the Main Zone pit. This contractor will be responsible for supplying, operating and maintaining all mining equipment, trucks and mining related infrastructure.

The mine is envisaged as an open pit operation using a conventional drill and blast process and conventional truck and shovel methods for material movement.

For the PEA, the life-of-mine (LOM) open pit mineable plant feed material within the conceptual pit designs is 9.22 Mt, inclusive of Measured, Indicated and Inferred resources, with a total waste movement of 3.24 Mt for an average stripping ratio of 0.35:1. With the inclusion of an additional 5.87 Mt of Inferred resource from the RAL legacy stockpile, the mine life is extended to 19 years. The mill feed rate used for the design is 806,000 t/y.

The economic parameters used as inputs for the mine optimization and design are summarized in Table 1.3.

Table 1.3
Pit Optimization Criteria East Kemptville Tin Project

Parameter	Unit	Value
Mining Cost (Mill Feed)	CAD/t mined	4.70
Mining Cost (Waste)	CAD/t mined	4.70
Legacy Stockpile Rehandle	CAD/t moved	1.25
Process Cost (Concentrator)	CAD/t mill feed	7.85
Process Cost (Sulphide Flotation)	CAD/t mill feed	0.63
G&A	CAD/t mill feed	1.54
Overall Pit Slope	degrees	48
Processing Recovery (Sn)	%	60.0
Metal Price (Sn)	USD/t	20,656
Treatment Charge	CAD/t conc	455
Transportation	CAD/t conc	225
Exchange Rate	USD to CAD	1.30

1.11 RECOVERY METHODS

The metallurgical process flowsheet for the Project is based on the mineral separation and recovery of a tin concentrate with a target grade of 55 wt.% Sn. A small portion of the copper, zinc, iron and indium will be collected into a sulphide concentrate which will be appropriately disposed of in the tailings facility unless a buyer for the material is found. Data gathered from both the SGS UK and Met-Solve metallurgical test programs along with historical information from previous operations and operating personnel was reviewed and used as the basis for developing the flowsheet.

Avalon’s objective is to construct a simple plant with as few unit operations as possible and focused purely on tin recovery. It is acknowledged that this approach will result in a lower than possible metal recovery, but it is believed that the low costs associated with such an approach will out-weigh any drop in recovery.

The PEA is based on the following assumptions derived from the testwork results:

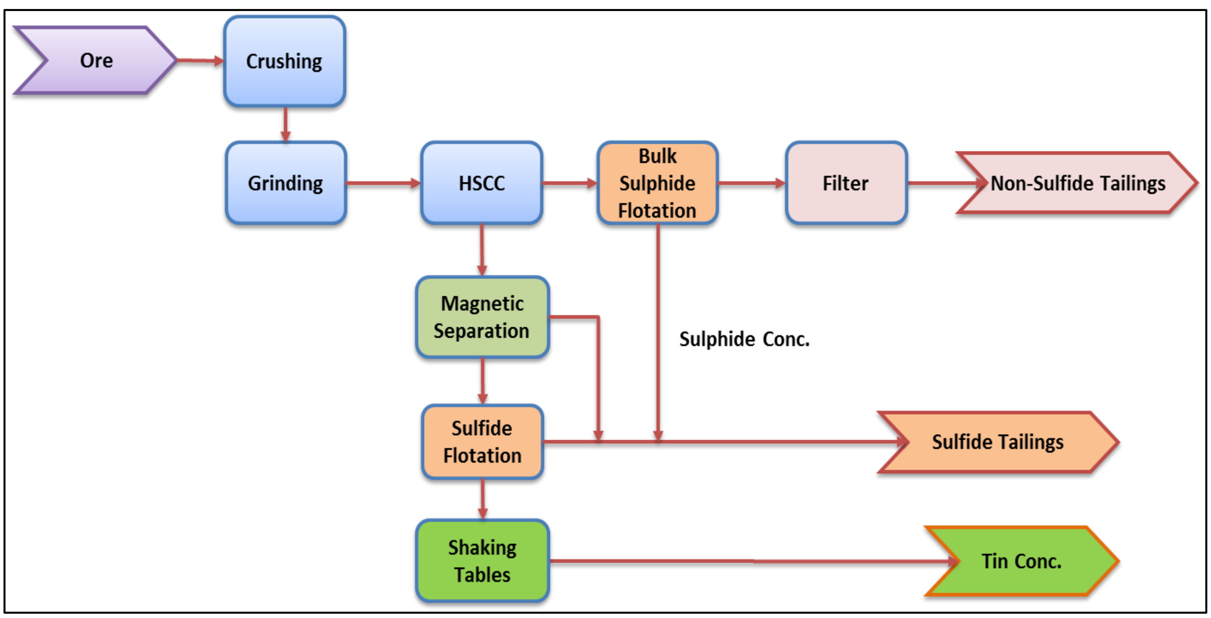
- 806,000 t/y of stockpiled mineralized material will be fed to the concentrator at a rate of 100 t/h.
- Target primary grind P_{80}= 80 Microns.
- The tin gravity concentrate grade of 55% Sn and tin recovery to concentrate of ~60%.
- Plant availability of 91.3% for the concentrator (8,000 h/y operating time).

- Preliminary tin recovery will be by centrifugal concentrators with shaking tables used to produce the final product.
- Concentrate cleaning will include magnetic separation and flotation to remove iron, copper and zinc sulphides.

A copy of the simplified flowsheet is included in Figure 1.1. The flowsheet consists of several conventional processes to produce a tin concentrate. This includes crushing, milling and classification, a series of gravity circuits using high-speed centrifugal concentrators (HSCCs), magnetic separation and flotation to remove the metal sulphide before going through a series of shaking tables. A bulk sulphide flotation circuit is also included in Year 6 of the operation to remove sulphides from the gravity tailings.

Benign flotation tailings will be filtered and used for capping the tailings facility. The bulk sulphides concentrate removed from the gravity tailings will be combined with the sulphide concentrate from the tin gravity circuit and stored under a cover of water to prevent oxidation.

Figure 1.1
Simplified Flowsheet



1.12 PROJECT INFRASTRUCTURE

Existing roads on site allow easy access to the entire site for operations and maintenance. A new haul road will be required from the mine pits to the processing plant. There is sufficient infrastructure in the area to support the labour force required for the project operations and no need is seen for accommodations at site.

1.12.1 Power, Fuel and Water

Primary power to the site will be provided by Nova Scotia Power via an existing line which will feed a new substation at site. Emergency/back-up power will be provided by a diesel generator.

Diesel storage and fueling stations will be provided on site for mobile equipment.

Raw water from the Tusket River will supply potable, fire and process water requirements. Process water will be recycled to keep make-up water requirements to a minimum. Process water will also be extracted from the 2 existing pits or recycled from the TMF in order to minimize raw water consumption and to also make the pits accessible for mining.

The existing water treatment facility which treats run-off from the tailing facility will be maintained for ongoing operations and modified as required to meet the new project demands, although minimal changes are anticipated to be required.

1.12.2 Buildings, Communication and Waste Handling

The intention is to erect a single pre-engineered and pre-fabricated building that can house the main processing plant (excluding crushing circuit), stores and workshop areas all under a single roof.

Proven, reliable and state-of-the-art telecommunications systems will be provided at the site for permanent operations and maintenance.

Waste materials (organic waste, hazardous and recyclable wastes, etc.) will be sorted on site and disposed of off-site using local contracting companies or existing municipal handling facilities.

1.12.3 Concentrate Storage and Shipping

Concentrate will be bagged, containerized and stored at site before shipment on a regular basis to the laydown area at the port in Shelburne or Halifax. On average, approximately 120 t of concentrate will be produced per month requiring the transportation of 4-5 truckloads per month from the site to the port.

1.13 MARKET STUDIES AND CONTRACTS

For the purposes of the PEA, Avalon has undertaken an in-house analysis of the markets for tin concentrates during the course of which it has consulted with industry participants and specialist consultants.

A tin price of USD21,038 /tonne has been used for the PEA, which is not only the World Bank forecast for 2020, but also is consistent with the LME price for tin during the first quarter of 2018 (USD21,187).

1.14 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Following the completion of an environmental baseline study, impact assessment and permitting, the East Kemptville mine operated between 1983 and 1992 at a production rate approximately 4 times higher than that envisioned for this project. The overall site is currently considered a brownfields site with ongoing perpetual treatment of runoff water.

The East Kemptville site has long term environmental liabilities that are the result of sulphide minerals that remain in the pit walls, low grade and waste rock stockpiles, and tailings, all of which generate acid mine drainage (AMD) to greater or lesser extents. At this time, these liabilities are being effectively managed by the surface rights holder through the collection, treatment and release of treated water.

An agreement between Avalon, the surface rights holder and the Government of Nova Scotia will be required, prior to development of this project, which details how and when Avalon will assume care and custody of the closed site. A letter describing this requirement was signed by the Ministry of Natural Resources (now Nova Scotia Energy and Mines).

The start of operations is not anticipated to be subject to approvals under the Canadian Environmental Assessment Act 2012 (CEAA) as the mine does not exceed any of the CEAA triggers, including mine and mill tonnages. The project is not anticipated to have any new impacts to terrestrial, fish or fish habitat, and will not impact any federally designated wildlife conservations areas. The project will be subject to the Nova Scotia Environment Act and associated regulations (including the Environmental Assessment Regulations), via the provincial “One Window” approach to mineral resource development chaired by Nova Scotia Energy and Mines.

Planned operations are an integral component of the overall mine rehabilitation strategy and to mitigate the present and ongoing sources of environmental liability. The brownfields site has known sources of AMD to both surface and groundwater. These are now well understood by Avalon and appropriate mitigations and closure plans identified for these historical impacts have been developed, as well as for any impacts anticipated from future operations.

Avalon is recognized for its leadership in Indigenous Engagement. It has already reached out to the Mi’kmaq First Nation to make them aware of recent small drill programs and to initiate dialog with them. Avalon has also initiated engagement with the local community.

1.15 CAPITAL AND OPERATING COSTS

1.15.1 Capital Cost Estimate

The estimated Project capital requirements are summarized in Table 1.4. All costs are reported as Canadian Dollars (CAD or $) with a base date of first quarter, 2018. It should be noted that, apart from the sulphide removal circuit in Year 5, provisions for what might normally be designated as "sustaining capital" are included in the operating costs.

The capital cost estimate for this Project is considered to be at a scoping level with an accuracy of +50%/-35% and carrying an average contingency of 18.6% on total initial estimated capital.

Table 1.4
Initial Capital Cost Estimate

Area	Capex CAD x 1,000	
	Initial Plant	Sulphide Removal (Year 5)
Mining	0	0
Concentrator	18,472	4,076
Tailings Disposal	544	0
Infrastructure	946	0
Total Direct Costs	**19,962**	**4,076**
EPCM	1,497	306
Freight & Transportation	861	188
Other Indirects	1,778	446
Total Indirect Costs	**4,136**	**940**
Owners Costs	1,000	500
Buildings & Tailings	750	100
Contingency	4,820	1,003
Total Capital Costs	**30,688**	**6,620**

Mining capital costs are assumed to be zero as the operation will engage a contract miner and all mining related capital costs are built into the contract mining operating costs.

Excluded from the pre-production capital cost estimate is the allowance for dewatering the two pits. This amount is estimated at CAD850,000, which increases the estimate to CAD31.5 million.

1.15.2 Operating Cost Estimate

A summary of the LOM average annual costs is presented in Table 1.5.

Table 1.5
Summary of Operating Costs

Category	Ave. Annual Costs (CAD'000)	CAD/t Milled	CAD/t Tin	CAD/t Conc.
Stockpile Reclaim & Mining	3,588	4.40	5,076	2,792
Concentrator Processing	6,556	8.04	9,274	5,102
Concentrate Transport	289	0.36	409	225
Remediation & Site Management	848	1.04	1,200	660
General & Administration	340	0.42	480	264
Total Production Costs CAD	**11,583**	**14.25**	**16,439**	**9,044**
Total Production Cost USD	**8,910**	**10.96**	**12,646**	**6,957**

1.16 ECONOMIC ANALYSIS

Micon has prepared this PEA of the Project on the basis of a discounted cash flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback and other measures of Project viability can be determined.

The technical parameters, production forecasts and estimates described elsewhere in this report are reflected in the base case cash flow model.

1.16.1 Macro-Economic Assumptions

An exchange rate of CAD1.30/USD is applied in the base case, approximately equal to current rates and to the trailing average over the past two years.

Micon has applied a real discount rate of 8% in its base case evaluation, approximating the weighted average cost of capital (WACC) for the Project.

The base case cash flow projection assumes a constant price of USD21,038/t tin metal.

Nova Scotia mining taxes, and Canadian federal and provincial income taxes payable on the Project have been provided for in the cash flow forecast.

No royalty has been provided for in the cash flow model.

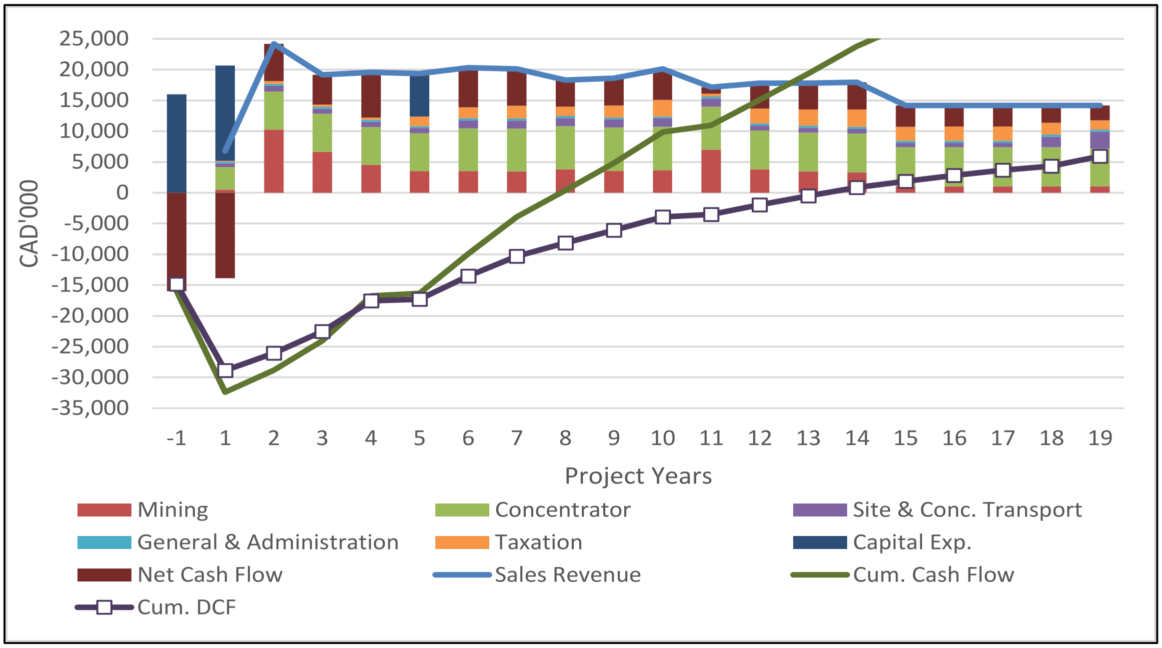
The base case Project annual cash flows are presented in Figure 1.2.

This PEA is preliminary in nature; it 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.

Before tax, the base case demonstrates an undiscounted payback period of 6.7 years, and an IRR of 15.0%. At an annual discount rate of 8%, the Project has a net present value (NPV_8) before tax of CAD17.8 million, and the payback period extends to 9.2 years.

After tax, the base case undiscounted payback period is 8.0 years, leaving a tail of 11 years planned production, and the Project has an IRR of 10.6%. The NPV_8 after tax is CAD5.6 million, and the payback period extends to 13.6 years.

Figure 1.2
Annual Cash Flow



1.17 Risks and Opportunities

The Project as currently envisaged presents the following risks and opportunities.

1.17.1 Head Grade to Mill

The opportunity presented by the drill hole spacing is that there may be areas of potential high-grade mining that are poorly defined and unrecognised at present due to the wide drill hole spacing thus increasing the mine life and financial return. The operating cost schedule provides CAD250,000 in each of Years 2 and 3 for conducting suitable drill programs within both pits once they are dewatered.

1.17.2 Resources

Opportunities exist to increase resources for the Project. This includes expanding the existing deposit resources as well as additional areas, such as the Duck Pond Zone and area west of the Baby Pit.

There has been no examination of the possibility of underground mining. Deep drilling on the Baby Zone has suggested that tin mineralization continues close to 100 m below the bottom of the presently planned pits. A detailed examination of this data may reveal underground mining potential in this and other areas of the property.

There are additional very low-grade stockpiles on surface which could potentially be processed if methods such as ore-sorting are demonstrated to have the ability to pre-concentrate the tin prior to the milling circuit.

1.17.3 Tin Price

An analysis of recent historical tin prices indicates that the LME listed price for tin has been above that value virtually continuously for more than the past 10 years. The LME listed price as of 1 May 2018 is USD21,395 and the World Bank Commodity Price forecast indicates tin a long-term price forecast of USD20,169 for 2025.

1.17.4 Tin Recovery

The tin recovery of 60% is based on the testwork program by Met-Solve, and Avalon believes that once the plant is up and running, this figure can be improved upon. With bench scale testwork, it is difficult to simulate the impact of recirculating streams and to optimize recovery over time so material that would be captured from such streams often reports to tailings during bench testing. With an operating plant, these streams are fully recycled, and operators have the opportunity to optimize recovery.

1.17.5 Mining

The forecast mining costs represent almost 30% of total production costs and are estimated using typical industry contractor rates for open pit operations of this size. Upon completion of the proposed drilling to update the resource model, further mine design work and haulage analyses are required before costing of the final tonnages of material (plant feed plus waste) to be mined can be more accurately defined.

1.17.6 Stockpile Grade

The grade of the material in the stockpile has been estimated by two surface sampling programs and by reviewing historical information, all of which produced similar results, and as a consequence an "inferred" resource has been determined by an external consultant. It is, however, planned to complete a drill program of the stockpile as soon as financing is available, partly to confirm the overall grade, but more importantly to map the internal grade distributions and produce a more representative schedule of feed grades shipped to the processing plant from this source.

1.17.7 Operating Life

The current operating life is 18.5 years; however, Avalon is confident that additional feed sources will be identified, and that the operating life will be extended.

1.17.8 Purchasing Used/Refurbished Equipment

The capital cost estimate has assumed all equipment is purchased new, but there are significant opportunities to reduce equipment costs, particularly for the crushers and mill, by purchasing used/refurbished items. Avalon is also aware of a number of used screens and gravity concentrators that could potentially be acquired.

1.17.9 Revenue from By-products

No provision has been made for up-grading the sulphide concentrate into marketable copper and zinc/indium concentrates for sale.

1.17.10 Foreign Exchange Rate

A lot of the mechanical equipment is being sourced from outside Canada and is priced in American dollars. Similarly, all revenue is in USD. An exchange rate of CAD1.30:USD1 has been used. Should the Canadian dollar strengthen this would be positive in terms of initial capex, but then negative with respect to subsequent revenue once in production.

1.17.11 Environmental Liability

By re-activating the Project, Avalon will be inheriting a number of (currently) long term environmental liabilities. However, by removing the low-grade stockpile, capping the tailings facility and depositing the balance of the tailings along with waste rock into the two pits, Avalon believes a "walk-away" closure strategy has been developed, eliminating these long-term liabilities.

1.18 CONCLUSIONS

Avalon has the opportunity to re-commence commercial tin production from the East Kemptville mine by establishing a small-scale operation processing an on-surface, low-grade stockpile and higher grade, near surface occurrences within the existing pits.

Avalon considers the tin concentrate produced (see Table 1.6) to be highly marketable. In early 2018, Avalon has entered into a non-binding MOU for the sale of all its forecast production with a well-known company that owns a large tin smelter. The formula used by this customer for determining concentrate pricing has been used by Avalon in the financial model.

Table 1.6
Final Tin Concentrate Analysis

Element	Sn	Cu	Zn	Fe	S	Pb	As	Cd
Value (%)	55.22	0.009	0.014	0.57	0.08	0.005	0.002	<0.0001
Element	**Ni**	**Co**	**Bi**	**Hg**	**Se**	**SiO_2**	**Mn**	**CaF_2**
Value (%)	0.006	<0.001	<0.0001	<0.0001	0.0001	9.04	0.35	0.55

The re-development model, as presently conceived, is an environmental remediation Project that will be financed through the sale of tin concentrates recovered in large part from previously-mined mineralized material on the site.

The Project enjoys strong support from the community as well as from local politicians, First Nations and environmental NGOs. Avalon is also in discussions with a number of local businesses towards collaboration on future opportunities including, among others, a long-term vision for re-development of the rehabilitated site.

The start of operations is not anticipated to be subject to approvals under the Canadian Environmental Assessment Act 2012 (CEAA) as the mine does not exceed any of the CEAA triggers including mine and mill tonnages. The Project will not have any new impacts to fish or fish habitat, nor will it impact on any Federal Wildlife Areas or Migratory Bird Sanctuaries. Final Permitting and Approval for the Project is therefore expected to be relatively short and simple.

1.19 RECOMMENDATIONS

The preliminary economic assessment presents an attractive Project and the opportunity to generate significant revenue for Avalon as well as remediating an environmental problem. It is recommended therefore that the Project continues to the next stage of development.

1.19.1 Recommendations for the Next Phase of Project Development

1.19.1.1 Resources

- The low-grade stockpile should be drilled, sampled and assayed to increase the confidence of the mineral resource estimate from an inferred category.
- Once de-watered, a program of infill drilling is recommended for the Main and Baby zones in order to improve the geological data base and to improve understanding of the controls on mineralization and variability of grade. Also, it is likely there are other areas of shallow, high grade material which could be added to the feed stock particularly if the tin price continues to trend upwards.
- During the course of operations, additional exploration should be conducted on other areas within and adjacent to the current property boundary in order to identify additional resources (e.g., Duck Pond area where prospective economic mineralization has already been identified).

1.19.1.2 Mining

- The mine designs and Project schedules should be completed to a more detailed level using the revised mineral resources resulting from the work recommended above.
- Mining contractors should be requested to provide a more detailed mining contract proposal using these updated detailed mine plans and schedules.
- The economic potential of mining deeper (either through open pit or underground methods) should be investigated for the Main and Baby Zone mineralization.

1.19.1.3 Processing Plant

- During the next phase of engineering, the proposed modular off-site fabrication and assembly philosophy should be adhered to as it will not only keep the up-front capital cost lower than normal but will also facilitate either future expansion or plant relocation to elsewhere once the East Kemptville resources have been exhausted.
- There is an opportunity to run a short pilot campaign to assess and optimize the initial "rougher" tin recovery performance. The purpose of this will be predominantly to fully optimize the grinding and classification circuit as minimizing over-grinding of the cassiterite is a key operating component. The rougher circuit operation will also provide an opportunity to optimize performance and confirm the expectation that a recovery >60% is achievable.
- The potential for using ore-sorting to upgrade the plant feed should be further investigated. This could have significant impacts on capital and/or operating costs either through the use of a smaller, cheaper processing plant or by significantly increasing the tin output through the same plant but over a shorter time frame. The pre-treatment by ore-sorting, of the "very low" grade stockpiles may also generate a suitably graded material to allow plant operations over a longer period.

1.19.1.4 Project Implementation

- The current 16-18-month implementation schedule is tight, and where possible, development activities should continue whilst Project funding is being secured. Such activities could include finalizing fixed equipment prices, confirming fabricators to be used and negotiating various service and supply contracts.
- There are various minor permitting studies which still need to be completed in order to gain site access for initiating construction activities. These studies should be completed as soon as possible in order to prevent any potential impact on the implementation schedule.
- Securing a final agreement with BHP still needs to be completed but this must be subject to finalizing a mutually beneficial transition arrangement to minimize Avalon's up-front exposure to the existing environmental liability.
- The start of operations is not anticipated to be subject to approvals under the Canadian Environmental Assessment Act 2012 (CEAA) as the mine does not exceed

any of the CEAA triggers including mine and mill tonnages. The Project will not have any new impacts to fish or fish habitat, nor will it impact on any Federal Wildlife Areas or Migratory Bird Sanctuaries. Final Permitting and Approval for the Project is therefore expected to be relatively short and simple.

1.19.2 Budget

The budget prepared by Avalon for the next phase of work to develop the East Kemptville Project towards production is presented in Table 1.7 below.

Table 1.7
Budget for the Next Phase of Project Development

Proposed Work	Estimated Cost (CAD)
Drilling and Resources Update	
Drilling Stockpile	250,000
Economic Study Update	
Mini Pilot Plant Trial	100,000
Preliminary Engineering and detailed cost estimates	300,000
Updated economic study and NI 43-101 report	100,000
Environmental	
General studies and permitting applications	100,000
Total Proposed Budget (all items)	**850,000**

Micon has reviewed Avalon's budget for the next phase of work on the East Kemptville Project and considers it to be reasonable.

2.0 INTRODUCTION

Micon International Limited (Micon) has been retained by Avalon Advanced Materials Inc. (Avalon) to prepare a Technical Report under Canadian National Instrument (NI) 43-101 which discloses the results of the preliminary economic assessment (PEA) for the East Kemptville Tin Project (East Kemptville Project), Yarmouth Co., Nova Scotia.

2.1 SCOPE OF THE REPORT

2.1.1 Mineral Resource Estimate

The mineral resource estimate on which this PEA is based is dated 7 May, 2018. This resource estimate was prepared by Avalon and is considered not to be materially different from the previous mineral resource by Hains Engineering Company Limited (Hains Engineering), which is described in an Avalon press release dated 31 October, 2014.

The mineral resource estimates in this PEA have been prepared in accordance with the requirements of Canadian securities laws, which differ from the requirements of United States securities laws. Unless otherwise indicated, all mineral resource estimates included in this PEA have been prepared following CIM Definition Standards in accordance with NI 43-101. The NI 43-101 is a rule developed by the Canadian Securities Administrators which establishes standards for all public disclosure an issuer makes of scientific and technical information concerning mineral projects. No reserves have been determined.

Canadian standards, including NI 43-101, differ significantly from the requirements of the United States Securities and Exchange Commission (the SEC), and reserve and resource information contained in this Technical Report may not be comparable to similar information disclosed by United States companies. In particular, and without limiting the generality of the foregoing, the term “resource” does not equate to the term “reserve”. Under the SEC standards, mineralization may not be classified as a “reserve” unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the reserve determination is made. The SEC’s disclosure standards normally do not permit the inclusion of information concerning “measured mineral resources”, “indicated mineral resources” or “inferred mineral resources” or other descriptions of the amount of mineralization in mineral deposits that do not constitute “reserves” by United States standards in documents filed with the SEC. United States investors should also understand that “inferred mineral resources” have a great amount of uncertainty as to their existence and as to their economic and legal feasibility. It cannot be assumed that all or any part of an “inferred mineral resource” exists, is economically or legally mineable, or will ever be upgraded to a higher category. Under Canadian rules, estimated “inferred mineral resources” may not form the basis of feasibility or pre-feasibility studies. Disclosure of the amount of minerals contained in a resource estimate is permitted disclosure under Canadian regulations; however, the SEC normally only permits issuers to report mineralization that does not constitute “reserves” by SEC standards as in-place tonnage and grade without reference to unit measures. The requirements of NI 43-101 for identification of “reserves” are also not the same as those of the SEC, and reserves reported by Avalon in compliance with NI 43-101 may not qualify as “reserves” under SEC standards. Accordingly, information concerning mineral deposits set forth herein may not be comparable with information made public by companies that report in accordance with United States standards.

2.1.2 Preliminary Economic Assessment

This PEA has been prepared by Micon under the terms of its agreement with Avalon. As discussed in the relevant sections of the report, Micon has prepared a mine plan and schedule, has reviewed the metallurgical testwork carried out on the property, the mineral processing flowsheet, has reviewed infrastructure requirements, prepared capital and operating cost estimates and an economic analysis of the Project.

Avalon owns the mineral rights to the East Kemptville Project and has been investigating various conceptual re-development plans and economic assessments to evaluate the potential for re-starting tin production at the mine.

Avalon initially investigated re-starting the East Kemptville tin mine in Nova Scotia based on the concept of returning the Project to its previous scale of production processing 10,000 tonnes per day (t/d) of material to generate tin, copper and zinc concentrates. Once it became apparent that the necessary capital for such an operation would likely prove difficult to source, Avalon continued the study assuming only the mining and processing of mineral resources contained within a low-grade stockpile and some shallow, relatively high-grade zones close to the surface of the two existing pits (the Main Zone Pit and the Baby Zone Pit). This new operation would target treating these plant feed sources at an annual rate of 806,000 tonnes to produce a tin concentrate only.

In addition to producing tin, the Project has expanded to include the key objective of remediation of the existing site through the treatment and sub-aqueous deposition of the acid generating Low Grade Stockpile and capping of the existing tailings facility with low sulphide tailings generated by the processing plant. This has the significant benefit of reducing the existing site environmental risk and long-term economic liability associated with the Project. The mining of the high-grade mineralization ensures subaqueous deposition of all tailings and waste rock as well as helps to ensure the economic return to incentivize investors, government and the present surface rights owner to support the Project.

The capital and operating cost estimates were developed from first principles and generated a production profile based on the mineralization available and metallurgical performance indicators from various testwork programs. This data was consolidated into a cash flow model which generated IRR and NPV figures used for assessing the likely economic viability and performance of the Project.

This report documents the findings and economic evaluation of Avalon’s East Kemptville Project.

2.2 QUALIFIED PERSONS AND SITE VISITS

The Qualified Persons for this Technical Report are the following:

- Richard Gowans, P.Eng.
- Christopher Jacobs, CEng., MIMMM
- Dayan Anderson, M.S., MMSA(QP).
- Jane Spooner, P.Geo.
- William Mercer P. Geo.
- Donald Hains P.Geo.
- Reid Smith. P.Geo.

2.2.1 Site Visits

Site visits have been carried out by the Qualified Person as follows:

William Mercer P. Geo.	Various occasions during 2014-2016
Donald H. Hains, P.Geo.	23-25 July and 3 September, 2014.

2.3 USE OF REPORT

This report is intended to be used by Avalon subject to the terms and conditions of its agreement with Micon. Subject to the authors' consent, that agreement permits Avalon to file this report as an NI 43-101 Technical Report on SEDAR (www.sedar.com) pursuant to Canadian provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this report, by any third party, is at that party's sole risk.

The requirements of electronic document filing on SEDAR necessitate the submission of this report as an unlocked, editable PDF (portable document format) file. Micon accepts no responsibility for any changes made to the file after it leaves its control.

The conclusions and recommendations in this report reflect the authors' best judgment in light of the information available to them at the time of writing. The Authors have relied on data available in published and unpublished reports, information supplied by the various companies that have conducted exploration on the property, and information supplied directly by Avalon. Micon has no reason to doubt its validity.

The authors and Micon reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to them subsequent to the date of this report. Use of this report acknowledges acceptance of the foregoing conditions.

Some of the figures and tables for this report were reproduced or derived from reports written for Avalon and Rio Algom Ltd (RAL).

2.4 UNITS

Units in this PEA are in the Système International d'Unités (SI), unless otherwise noted.

Cost estimates and other inputs to the cash flow model for the Project have been prepared using constant money terms, i.e., without provision for escalation or inflation. All costs are presented in Canadian dollars (CAD) unless otherwise noted. Prices for metals are given in United States dollars (USD) unless otherwise noted.

The CAD:USD exchange rate assumption of 1.3:1 has been used for cost estimates and the financial analysis.

2.5 LIST OF ABBREVIATIONS

Table 2.1 provides a list of the abbreviations used in this report.

Table 2.1
List of Abbreviations

Abbreviation	Term
°	Degree(s)
°C	Degree(s) Centigrade
°F	Degree(s) Fahrenheit
<	Less than
>	Greater than
µm	Micrometre(s) (micron = 0.001 mm)
%	Percent, percentage
%RSD	Percent relative standard deviation
A	Ampere(s)
AA	Atomic absorption
ABA	Acid Base Accounting
Actlabs	Activation Laboratories Ltd.
Ag	Silver
ALS	ALS Canada Ltd.
AMD	Acid mine drainage
ANFO	Ammonium Nitrate and Fuel Oil
As	Arsenic
Avalon	Avalon Advanced Material Inc.
B	Billion
BMA	Bulk mineral analysis
BQ	Diamond drill core size 36.5 mm (inside diameter of core tube)
Btu	British thermal units
C	Carbon
ca.	Circa, approximately
CAD	Canadian dollar(s)
CAD/t	Canadian dollars per tonne
CANMET	CANMET Mining and Mineral Sciences Laboratories
CEAA	Canadian Environmental Assessment Act
CDN	CDN Resource Laboratories Limited
cfm	Cubic feet per minute

Abbreviation	Term
CIF	Cost insurance freight
cm	Centimetre(s)
CNF	Cost and freight
CTP	Coarse Tailings Pile
Cu	Copper
CV	Coefficient of variation
d	Day(s)
DDH	Diamond drill hole
DLC	Davis Lake Complex
DMS	Dense media separation
dmt	Dry metric tonne(s)
dtpd	Dry metric tonnes per day
dwt	Dead weight tonne(s)
d/y	Days per year
EKL	East Kemptville leucogranite
EKSZ	East Kemptville Shear Zone
EMP	Electron microprobe
ESIA	Environmental and Social Impact Assessment
F	Fluorine
Fe	Iron
FOB	Free On Board
ft	Foot, feet
ft^3	Cubic foot, feet
FW	Footwall
g	Gram(s)
g	Acceleration due to gravity
g/cm^3	Grams per cubic centimetre
g/L	Gram(s) per litre
g/t	Gram(s) per tonne
GA	General Arrangement
gal	Gallon(s)
GHG	Green House Gas (emissions)
gpm	Gallons per minute
GPS	Global positioning system
GWh	Gigawatt-hour(s)
H	Hydrogen
h	Hour(s)
h/d	Hours per day
ha	Hectare(s)
HCl	Hydrochloric acid
HDPE	High density polyethylene
HF	Hydrofluoric acid
HIMS	High intensity magnetic separation
HP	Horsepower
HQ	Diamond drill core size 63.5 mm (inside diameter of core tube)
HSCC	High-speed centrifugal concentrators
Hz	Hertz
IP	Induced polarization
ICP	Inductively coupled plasma
ICP-AES	Inductively coupled plasma atomic emission spectroscopy
ICP-MS	inductively coupled plasma mass spectrometry

Abbreviation	Term
ICP-OES	inductively coupled plasma optical emission spectrometry
ID^2	Inverse distance squared
in	Inch(es)
In	Indium
INAA	Instrumental neutron activation analysis
IRR	Internal rate of return
ITA	International Tin Association
ITRI	International Tim Research Institute
K	Potassium
k	Kilo (thousand)
kg	Kilogram(s)
kg/h	Kilograms per hour
kg/m^3	Kilograms per cubic metre
kg/t	Kilograms per tonne
km	kilometre(s)
km/h	Kilometres per hour
kPa	Kilopascal(s)
kV	Kilovolt(s)
kVA	Kilovolt-ampere
kW	Kilowatt(s)
kWh	Kilowatt hour(s)
kWh/t	Kilowatt hours per tonne
L	Litre(s)
L/s	Litres per second
LAN	Local area network
lb	Pound(s)
lb/ft^3	Pounds per cubic foot
LCT	Locked cycle test
LHD	Load-haul-dump
LIMS	Low intensity magnetic separator
LME	London Metal Exchange
LOI	Loss on ignition
LOM	Life of mine
M	Mega (million)
m	Metre(s)
m^3	Cubic metre(s)
Mm^3	Million cubic metres
Ma	Million years
m/min	Metres per minute
m/s	Metres per second
mA	Milliampere(s)
masl	Metres above sea level
Mg	Magnesium
mg/L	Milligrams per litre
MIBC	Methyl isobutyl carbinol
min	Minute(s)
ML	Million litres
mm	Millimetre(s)
MMI	Mobile Metal Ion
Mo	Molybdenum
MOU	Memorandum of Understanding

Abbreviation	Term
MPa	Megapascal(s)
Mt	Million tonnes
MW	Megawatt(s)
MWh	Megawatt hour(s)
n.a.	Not applicable
Na	Sodium
NAD	North American Datum
NAG	Net acid generating
NDB	Non directional beacon
NGO	Non-Government Organization
NI 43-101	Canadian National Instrument 43-101
NN	Nearest neighbour
NPV	Net present value
NTS	National topographic system
NSDNR	Nova Scotia Department of Natural Resources
NSR	Net smelter return
NQ	Diamond drill core size 47.6 mm (inside diameter of core tube)
ORE	Ore Research & Exploration Pty Ltd.
oz	Ounce(s)
PAG	Potentially acid generating
P&ID	Process and instrument diagram
Pa	Pascal(s)
Pa.s	Pascal-second
PAX	Potassium amyl xanthate
Pb	Lead
PMA	Particle mineral analysis
ppb	Parts per billion
ppm	Parts per million
QA	Quality assurance
QA/QC	Quality assurance/quality control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning electron microscopy
QC	Quality control
RAL	Rio Algom Limited
RC	Refining charge
RMA	Reduced major axis
RNAV	Random area navigation
ROM	Run-of-mine
RQD	Rock quality designation
RSD	Relative standard deviation
s	Second(s)
S	Sulphur
SAG	Semi-autogenous grind
Sb	Antimony
SD	Standard deviation
SEM	Scanning electron microscope
SG	Specific gravity
SI	International system of units
Si	Silicon
SIPX	Sodium isopropyl xanthate
SMB	South Mountain Batholith
Sn	Tin

Abbreviation	Term
Sn(eq)	Tin equivalent
t	Tonne(s) (metric = 1,000 kg)
t/d	Tonnes per day
t/h	Tonnes per hour
t/y	Tonnes per year
Ta	Tantalum
TC	Treatment charge
TMF	Tailings Management Facility
TSS	Total suspended solids
UCS	Unconfined compressive strength
VHF	Very high frequency
VOR	VHF omni-directional range
WACC	Weighted average cost of capital

3.0 RELIANCE ON OTHER EXPERTS

Micon has reviewed and analyzed data provided by Avalon and its consultants and has drawn its own conclusions therefrom. Micon has not carried out any independent exploration work, drilled any holes or carried out any sampling and assaying on the property.

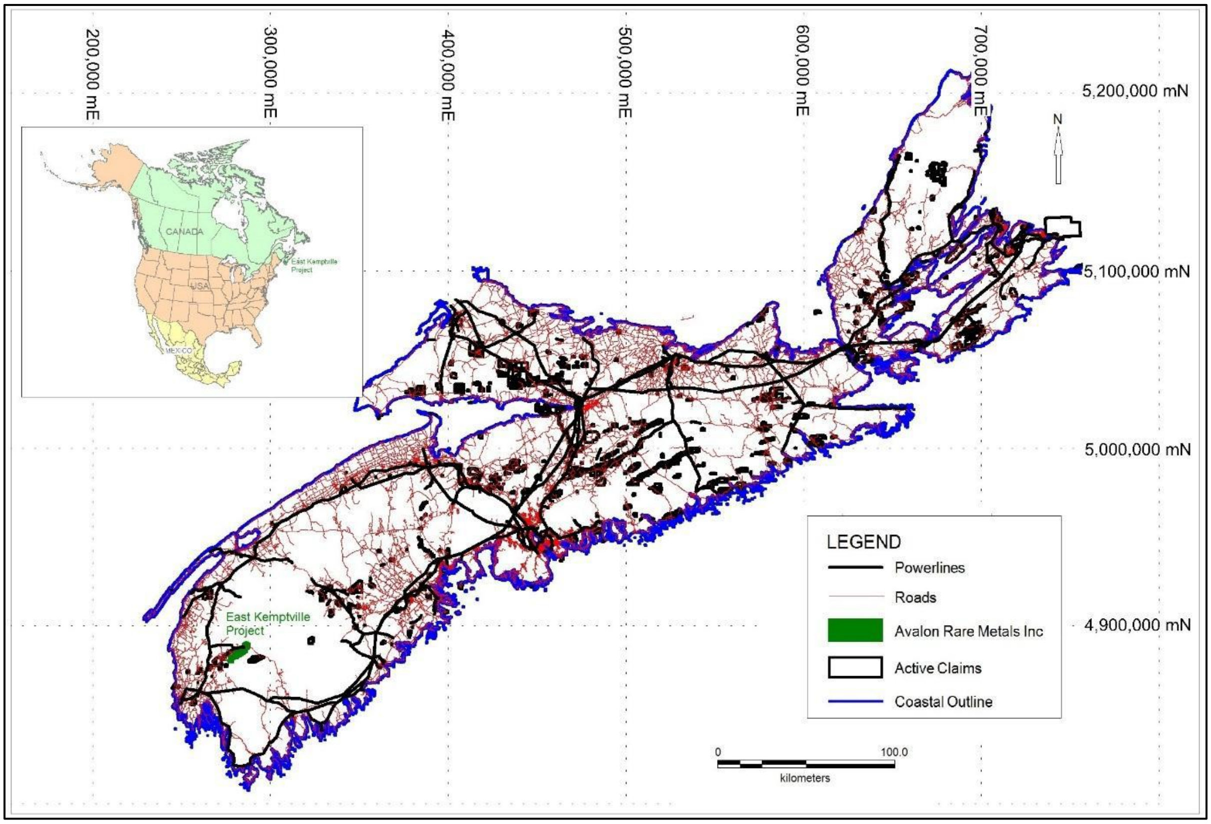
While exercising all reasonable diligence in checking, confirming, and testing it, Micon has relied upon Avalon's presentation of the Project data from previous operators and from Avalon's exploration experience at the East Kemptville Project in formulating its opinion.

Micon has not reviewed any of the documents or agreements under which Avalon holds title to the East Kemptville Project or the underlying mineral concessions and Micon is not qualified to comment as to the validity of the mineral titles claimed. A description of the properties, and ownership thereof, is provided for general information purposes only. The existing environmental conditions, liabilities and remediation have been described where required by NI 43-101 regulations. These statements also are provided for information purposes only and Micon is not qualified to comment in this regard.

4.0 PROPERTY DESCRIPTION AND LOCATION

The East Kemptville tin-indium Project is located on NTS map sheet 21A/04A and /05B in Yarmouth County, southwestern Nova Scotia. The property is located approximately 180 km southwest of Halifax, the provincial capital, and 55 km northeast of the town of Yarmouth, a port with population of approximately 7,500 (see Figure 4.1).

Figure 4.1
Property Location Map

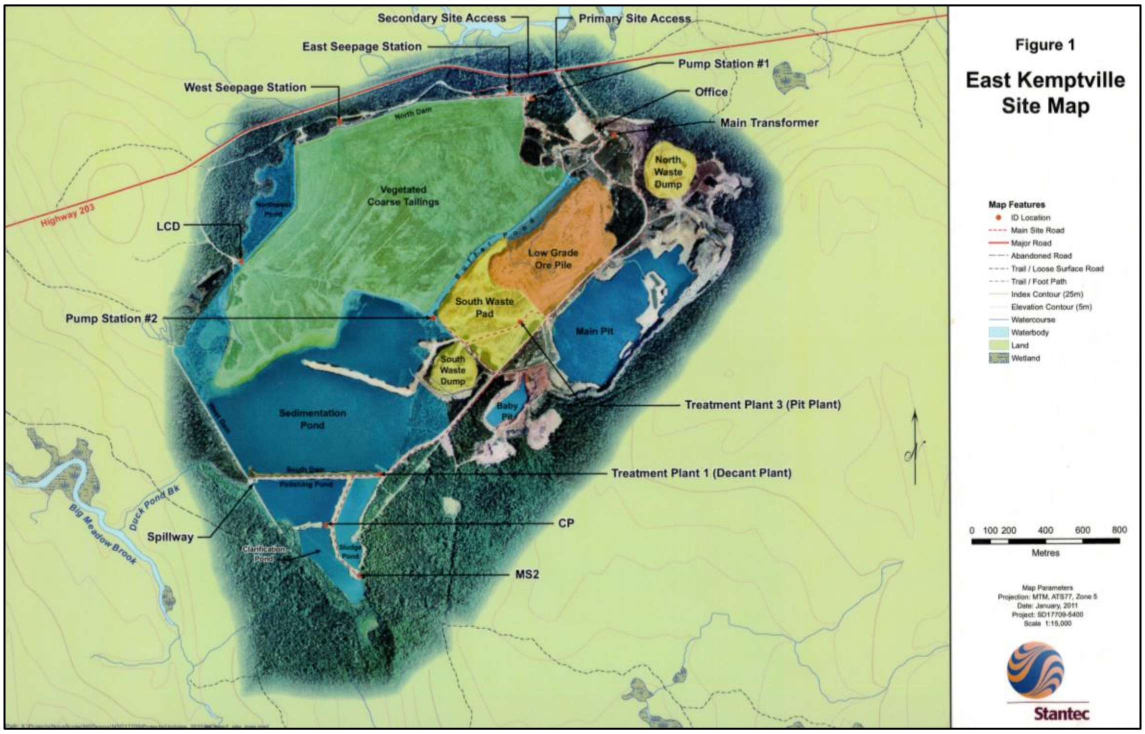


Source: Avalon, 2015.

Highway 203, constructed in the 1800s to facilitate the transportation of metal concentrates to the port of Shelburne, crosses the property and links it with Shelburne and the port of Yarmouth, both of which are on the Atlantic Ocean. Yarmouth is located some 45 km to the southeast of the property. The highway is in reasonable condition eastwards from the junction with Highway 101 but requires upgrading from the old mine entrance southwards to Shelburne which is the intended port of shipment for concentrates from East Kemptville.

Site facilities at the East Kemptville property are shown in Figure 4.2.

Figure 4.2
Site Facilities Map



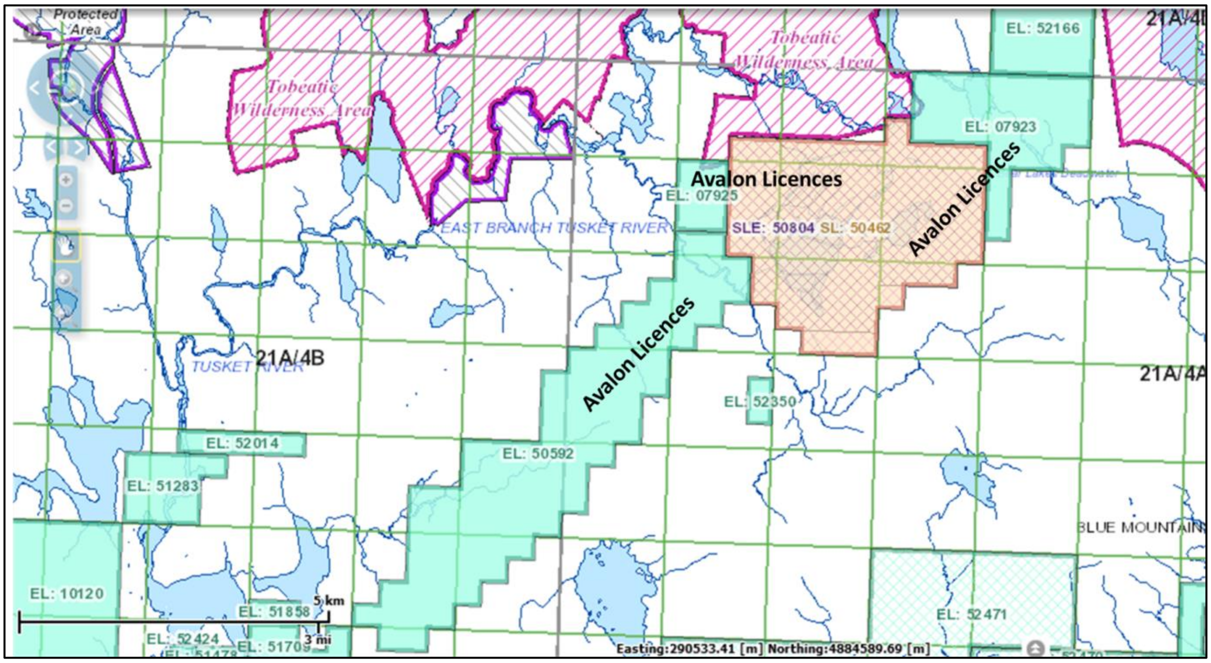
Source: Stantec Report, 2016.

4.1 PROPERTY TENURE

Avalon holds a 100% interest in the property via Special Licence 50462, which combines Special Licence 1/12 (now expired) and supersedes Application 40032. The area covered by Special Licence 50462 includes the Closure Area represented by the former East Kemptville Tin Mine property, which is currently under the management of RAL, the surface rights holder.

Figure 4.3 shows the location of Special Licence 50462.

Figure 4.3
East Kemptville Property Claim Map



Source: Avalon, 2018.

The details of the claims within Special Licence 50462 are summarized in Table 4.1.

Table 4.1
East Kemptville Tin-Indium Project – Claims within Special Licence 50462

Map Number	Tract Number	Special Licence 50462	
		Claim Designation	**Units**
21A/4A	81	M,N,O,P	4
21A/4A	82	A,B,C,E,F,G,H,J, K,L,M,N,O,P,Q	15
21A/4A	83	J,Q	2
21A/4A	86	A,B,G,H,J,K,P,Q	8
21A/4A	87	A to Q	16
21A/4A	88	A to Q	16
21A/4A	105	A,B,C,D,E	5
21A/4A	106	A,B,C,D	4
21A/4A	107	A,B	2
Total Claim Units			**72**
Ha			**1,165.5**
Acres			**2,880**

Except for claims 81M and 82A, B and C, the property is located on the site of a former tin mine with existing closure and reclamation activities carried out by the private surface rights owner. Given the closure and reclamation activities at the site, the lands were deemed withdrawn by the Government of Nova Scotia and exploration rights permitted only by way of a special licence or special lease granted by the Minister with the approval of the Governor in Council in accordance with Section 22 of the Mineral Resources Act, SNS 1990, c 18 (the East Kemptville Closure Area). The East Kemptville Closure Area is that property withdrawn by the Government of Nova Scotia, with the boundaries clearly indicated on the mineral licence maps of the government NOVAROC online map system.

Pursuant to Section 22 of the Mineral Resources Act, the mineral rights covering a portion of the East Kemptville Closure Area were first acquired by Avalon (then Avalon Ventures Ltd.) as Special Licence 1/06 dated 1 August, 2006 and approved by Order in Council on 21 September, 2006 (Special Licence 1/06). Special Licence 1/06 was for a three-year term ending 31 July, 2009, with allowance for two subsequent renewals of one year each with the approval of the Minister of Natural Resources. Special Licence 1/06 covered the area described in NTS 21A4A, tracts 81, 82, 87, 88 for a total of 880 acres or 356 ha. Renewal was contingent on Avalon undertaking work acceptable to the Minister. The work program for Special Licence 1/06 totalled CAD2,250,000 to be completed by 31 July, 2009.

Due to time required to achieve agreement with RAL in order to gain access to the site, Avalon was granted a renewed mineral licence from the Government of Nova Scotia in the form of Special Licence 1/12 dated 14 December 2012. Special Licence 1/12 covered the same area as Special Licence 1/06 and included the same expenditure requirements. Special Licence 1/12 had a two-year term commencing 1 October, 2012 and ending 30 September, 2014. Due to extended access negotiations, the Government of Nova Scotia granted an extension for submitting work statements to 1 November, 2014.

As of 24 April, 2015, the Government of Nova Scotia issued Special Licence 50462 to Avalon covering the entire East Kemptville Closure Area. The new Special Licence is for a term of three years, renewable twice for one year. While Avalon's Special Licence 50462 is active, the Mineral Resources Act provides protection against competing applications to parties with active applications under consideration. (See Mineral Resources Act, Sections 22(6) which provides, "*No application for a special licence shall be accepted for areas upon which another application for a special licence is under consideration*" and Section 30 which provides, "*No application for an exploration licence shall be accepted for areas that are subject to an exploration licence, special licence, non-mineral registration, lease, special lease or application for any of them, unless the applicant holds the mineral right or non-mineral registration*.")

As of the date of this report, the Special Licence 50462 has been renewed by the Government of Nova Scotia to 2 February 2019. Note that the pending application by Avalon for a Mineral Lease in the same area is recorded on the official Nova Scotia claim maps online system with the note of application SLE50804.

4.2 SURFACE RIGHTS AND ACCESS

The surface rights overlying Special Licence 50462 are a mix of private and Crown ownership. A portion of the Crown lands have been leased to RAL. In order for Avalon to gain access to the lands to conduct field operations, consent of the Government of Nova Scotia and RAL is required.

Avalon and RAL reached agreement on access terms and signed an Access Agreement as of 14 May, 2014 allowing Avalon to conduct the 2014 drilling program. The agreement expired on 30 September, 2014 and was renewed for the 2015 drilling and other field operations.

Consent to access Crown lands was obtained from the Government of Nova Scotia pursuant to the Permit for Mineral Exploration on Crown Land No. 60A 14 ME01 dated 16 July, 2014 for the 2014 drilling season and Permit 60A 15 ME01 (amended) for the 2015 drilling season. These permits required Avalon to post an environmental performance bond for site disturbances related to the drill programs. The 2014 permit was complied with and Avalon's reclamation activities after the drill program were approved by the Government of the Province of Nova Scotia and the performance bond was refunded in September, 2014. A further bond for CAD5,000 was required and posted in 2015 to cover that drill season.

4.3 OTHER MINERAL CLAIMS

In addition to Special Licence 50462, Avalon has a 100% ownership of a further three exploration licences, 7923, 7925, and 50592 (see Table 4.2) totalling 132 claims that surround and extend southwest from the area encompassed by Special Licence 50462. These licences cover 2,136.82 ha (5,280 acres). As of the date of this report, all these claims are in good standing.

Table 4.2
Avalon Exploration Licences Adjacent to East Kemptville (as of August, 2018)

Map Number	Exploration Licence	Tract Number	Claim Designation	Units	Ha	Acres	Expiry Date
21A4A	7923	89	E, M, N	3	48.56	120	
21A4A	7923	104	B, C, D, E, F, G, H, J, K, L, M, N, O, P, Q	15	242.82	600	
21A4A	7923	105	F, G, H, J, K, L, O, P, Q	9	145.69	360	
21A4D	7923	8	A, B, C, D, E, F, G, H, M, N	10	161.88	400	**01-Dec-18**
21A4D	7923	9	A, B, C, D, F, G, H, J, K, L, P, Q	12	194.26	480	
Subtotal	**7923**			**49**	**793.21**	**1,960**	
21A4A	7925	86	E, F, L, M, N, O	6	97.13	240	
Subtotal	**7925**			**6**	**97.13**	**240**	**01-Dec-18**
21A4B	50592	26	O, P	2	32.38	80	
21A4B	50592	47	A, B, C, D, F, G, H, J, K, P, Q	11	178.07	440	
21A4B	50592	48	E, L, M, N, O, P	6	97.13	240	
21A4B	50592	49	A, B, C, D, E, F, G, H, J, K, L, M, N, O, P, Q	16	259.01	640	**19-Sep-18**
21A4B	50592	50	A, B, G, H	4	64.75	160	
21A4A	50592	60	E, M, N, O	4	64.75	160	
21A4A	50592	61	B, C, D, E, F, G, J, K, L, M, N, O, P, Q	14	226.63	560	
21A4A	50592	72	A, H, J	3	48.56	120	

Map Number	Exploration Licence	Tract Number	Claim Designation	Units	Ha	Acres	Expiry Date
21A4A	50592	84	A, B, C, G, H, J	6	97.13	240	
21A4A	50592	83	D, E, F, K, L, M, N, O, P	9	145.69	360	
21A4A	50592	86	C, D	2	32.38	80	
Subtotal	**50592**			**77**	**1,246.48**	**3,080**	
Grand Total				**132**	**2,136.82**	**5,280**	

4.4 OPERATING PERMITS

In order to complete the 2014 and 2015 drill programs on the property, Avalon was required to file a "Notice to Drill" with the Government of Nova Scotia. As noted above, the permits were granted for the drill programs as proposed with the requirement for posting an environmental performance bond, which Avalon posted with the government. As all performance bonds associated with diamond drill programs in 2014 and 2015 have been returned by the government, Avalon has no outstanding environmental obligations on the property as of July, 2017.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The East Kemptville site is located 55 km by road east of the town of Yarmouth, Nova Scotia, and accessed from Yarmouth via Nova Scotia paved Highways 340 to Carleton and then 203 to the site. Yarmouth lies on Highways 103 and 101, approximately 300 km by road from Halifax.

Highway 203 extends southeast to the town of Shelburne, where it connects to Highway 103 leading to Halifax and Yarmouth. When the East Kemptville mine was in operation, some concentrates were taken by road to Shelburne, a distance of approximately 55 km, where the concentrate was transferred to ships in Shelburne harbour. Shelburne harbour is among the largest natural harbours in the world, being 10 nautical miles long, with depths ranging from 8 m at low tide to 14.6 m at high tide. The harbour is open year-round.

An existing air strip near Yarmouth can be used for access from outside the area and discussions are underway regarding the possibility of re-introducing commercial flights to Yarmouth to support the Project. It is classed as an international airport with customs and immigration facilities. The airport has two asphalt-surface runways, one 5,000 ft (1,525 m) long, the other 6,000 ft (1,830 m) long, and is capable of handling B737/A320 equivalent aircraft.

5.2 CLIMATE

The nearest weather station to the East Kemptville site is Yarmouth. The climate may differ somewhat as East Kemptville is about 55 km inland.

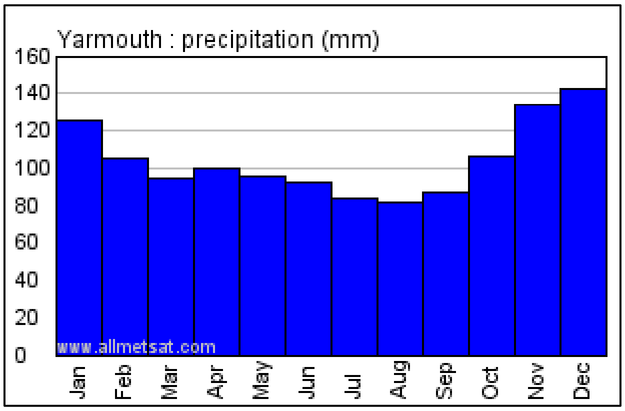
The Closure Plan for the mine site provides some details on climate. The text below is derived from this and from the websites www.weatherspark.com and www.allmetstat.com which publish a number of graphs of climatic variables. Records are available for the Yarmouth Airport weather station over the course of an average year, based on historical records from 1982 to 2012. The Closure Plan states:

> *"The regional climate is modified continental, resulting from marine interactions with predominantly west to east continental air masses. Influxes of moist Atlantic air may produce mild spells in winter and cool weather in summer. Measurable precipitation days vary from about 160 near Yarmouth to about 100 to 120 days inland. Snowfall accounts for about 13% of precipitation"* (RAL, 1993).

Precipitation is seasonal with a winter maximum and late spring/early summer minimum (See Figure 5.1). Annual total precipitation is in the 1,200 to 1,400 mm range. Median snow cover duration varies from 100 days along the coast to 130 days inland. Based upon a small period of record, the 24-hour rainfall amount for a 50-year storm is about 104 mm and for a

100-year storm is about 111 mm. The frequency of occurrence of hurricane remnants, and the short period of record (1982-2012) means that these figures provide a guideline only.

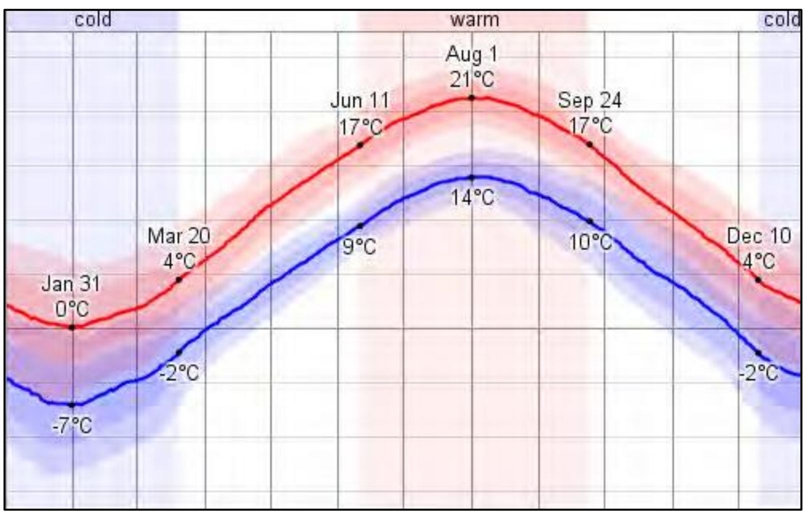
Figure 5.1
Monthly Average Precipitation



Source: www.allmetstat.com.

Mean daily temperatures range from about 17°C to 21°C in mid-summer to approximately -3° to -4°C in mid-winter and are illustrated in Figure 5.2. The range of about 23°C is much wider than that found in a true maritime climate.

Figure 5.2
Daily Average High and Low Temperatures for Yarmouth Airport

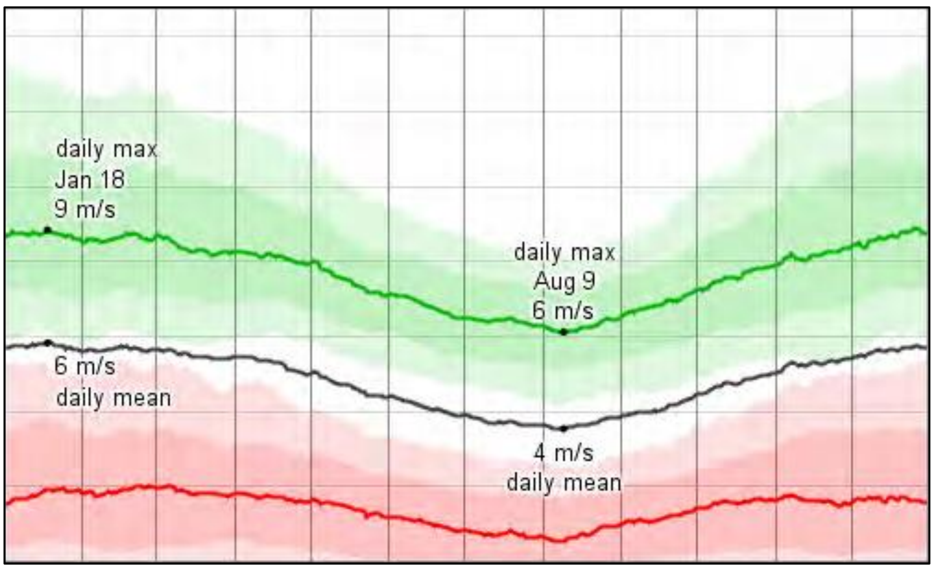


Daily average high = red, daily average low = blue.
Source: www.weatherspark.com.

These values can be compared to those contained in the baseline data records at the East Kemptville mine site for the period 1990-1991, where mean daily temperatures range from about 18°C to 19.5°C in midsummer to approximately 0.6°C to -1.5°C in mid-winter. The mean (50%) frost free period varies between 106 and 171 days and extends from about mid-May through September (RAL, 1993).

Over the course of the year, typical wind speeds vary from 1 m/s to 9 m/s (light air to fresh breeze), rarely exceeding 13 m/s (strong breeze) (See Figure 5.3). The highest average wind speed of 6 m/s (moderate breeze) occurs around 18 January, at which time the average daily maximum wind speed is 9 m/s (fresh breeze). The lowest average wind speed of 4 m/s (gentle breeze) occurs around 9 August, at which time the average daily maximum wind speed is 6 m/s (moderate breeze).

Figure 5.3
Average Daily Wind Speed with Percentile Bands for Yarmouth Airport



Minimum = red, Maximum = green, Average = black.
Inner Band from 25th to 75th percentile; outer band from 10th to 90th percentile.
Source: www.weatherspark.com.

Average wind directions indicate that winds from the north dominate in winter (55%) and from the south in summer (60%). The wind is roughly 50% of the time from the east or west.

Exploration and mining operations may be carried out year-round.

5.3 LOCAL RESOURCES

The town of Yarmouth has basic requirements for operations and would be, along with Shelburne, the likely location for employees to reside, as well as the rural areas and small communities in between. Yarmouth had its highest population in 1961 at 8,636 and at present is believed to be of the order of 6,700. The population has been dropping steadily in recent years due to lack of economic opportunity.

The main economic activity of the community is lobster fishing though the Nova Scotia government is attempting to promote tourism.

5.4 INFRASTRUCTURE

The Project site is well provided with infrastructure:

1. Power is available on site with a 69 kV power line served by Nova Scotia Power, a subsidiary of Emera, which is a Nova Scotia-based electrical power utility listed on the Toronto Stock Exchange. The substation remains intact but is not currently online.
2. Gate house and administrative building in good condition.
3. A well-developed internal road network at the mine site.
4. Ample water supply is available, given the presence of rivers, lakes and rainfall. Water will also be extracted from the 2 existing pits and recycled from the TMF.
5. The highway to the site is paved, but in poor condition, and would require rebuilding for operations. Redevelopment of the mine operation is expected by Avalon to be viewed favourably by the provincial government with respect to rebuilding Highway 203.
6. There is no shortage of local housing, or land for housing, for employees.
7. Tailings facilities and ponds and related water pipelines and pumping infrastructure remain in place and are in excellent condition.
8. Two lime silos and associated mixing and dosing equipment are on site. These facilities are currently maintained for treatment of mine water runoff. One silo is actively operated, while the second is on stand-by duty and operated on an as-required basis.
9. The existing Tailings Management Facility (TMF) currently on-site was originally designed for 60 million tonnes (Mt), but this includes most of the tailings deposited above the water table where it is exposed to oxidation and the generation of Acid Mine Drainage (AMD). The subaqueous capacity of the TMF, allowing for a 1 m water cover to prevent oxidation, is 1.75 million m^3 (2.7 Mt of tailings). This is sufficient capacity for the first 3 years of the proposed operating scenario.

5.5 PHYSIOGRAPHY

The topography of the immediate project area is subdued, with small hills with elevations varying from 60 masl to 130 masl in the highest areas. The only water bodies within the project area are man-made relating to the previous mining activities, being the pit lakes and the tailings pond. The area is either covered in outcrop or glacial till, which locally averages about 10 m thick.

The area was extensively glaciated, and the main Wisconsin ice advance over Nova Scotia moved southerly and terminated at the edge of the continental shelf about 300 km south of the present shoreline. During the retreat of the Wisconsin ice sheet, a secondary ice cap developed over the area underlain by the South Mountain Batholith and produced a radial flow outward in all directions away from this centre. In southwestern Nova Scotia, the direction of ice movement varied from 120° to 170° (Kohlsmith, 1984).

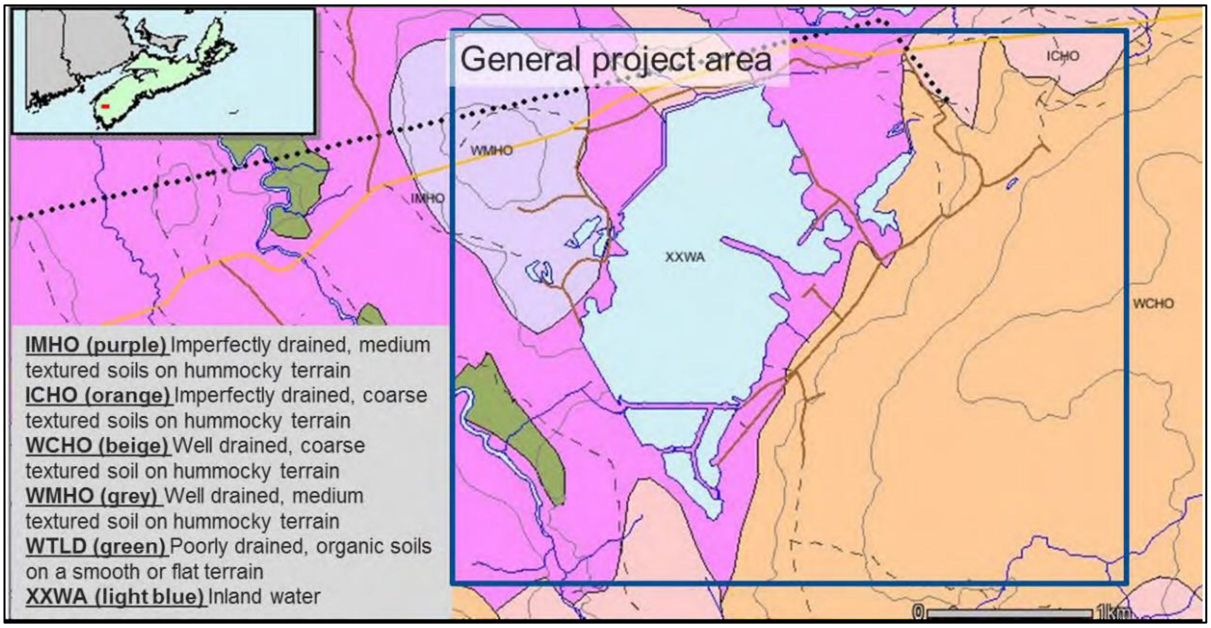
Virtually the whole of the leased area is mantled with a layer of coarse sandy, stony till of uneven thickness. Large surface boulders (erratics) are common in many areas with concentrations occurring in eastern and southern portions of the study area. Detailed data on till compositions are available from the geotechnical exploration drilling programs. In nearly all instances, tills are composed of varying proportions of silty sand, sand, gravel, cobbles and boulders. Concentrations of boulders and cobbles are generally greatest in surface layers on upland sites where the differential erosion has removed finer materials. In extreme cases, boulders comprised more than 50% of surface and near surface materials. Concentrations of silty sands and sands are common in depressions and lowland sites where they are frequently overlain by organic deposits (RAL, 1993).

5.6 VEGETATION

The Project site is within the Western Ecoregion of Nova Scotia (Ecological Land Classification for Nova Scotia, Report DNR 2005). Within that classification, the dominant landform is “hummocky terrain” as noted in Figure 5.4.

Forest stands of red spruce, hemlock and white pine are most prominent in the Western Ecoregion and perhaps more so than anywhere else in the province. Stands of this distinctive Maritime forest occur on the sandy and generally shallow soils of the ecoregion. Other dominant trees include red oak and red pine. Pure stands of white pine can be found on the drumlins, eskers and flutes of the barren lands. Although balsam fir occurs in most of the forest types, its dominance within stands has been reduced by the damaging effects of the balsam woolly adelgid (*adelges piceae*) a gout-causing forest pest introduced from Europe circa 1910. Significant portions of the ecoregion are occupied by stunted forests of black spruce on the bogs. Large tracts of red maple occur on other wetlands associated with the western rivers.

Figure 5.4
Land Classification in the Project Area



Note: The tailings ponds appear on this government map, but not the pit lakes.
Source: Government of Nova Scotia, Department of Natural Resources. Ecological Land Classification Map of Nova Scotia, online at www.gis4.natr.gov.ns.ca.
http://gis4.natr.gov.ns/ca/website/nselcmap/viewer.htm.
The explanation for the map is given at: http://novascotia.ca/natr/forestry/ecological/pdf/ELCrevised2.pdf.

6.0 HISTORY

Significant, greisen-style tin mineralization was discovered in granitic outcrop in the East Kemptville area, in 1978, by Shell Canada Resources Limited (Shell) as a direct result of a large follow-up regional geochemical till sampling program carried out in southwestern Nova Scotia. This program was in part driven by the 1976 discovery of highly mineralized, meta-sedimentary rocks in the Plymouth quarry some 20 km to the southeast by a group of local prospectors led by Avard Hudgins and Merton Stewart, as well as from a subsequent airborne radiometric survey released by the Geological Survey of Canada in 1977.

Shell initially drilled a total of 136 BQ diamond drill holes for a total of 12,450 m during the period March, 1979 through August, 1980. Holes were mainly drilled on 50 m by 100 m centres at -450 grid east (1,200) normal to the strike of the intrusive contact. A further 23 diamond drill holes totalling 1,840 m were drilled on 25 m centres in a 100 m by 100 m cell in the centre of the deposit to test for grade continuity between existing holes. Subsequently, an additional four diamond drill holes totalling 490 m were drilled as part of an underground exploration and bulk sampling program conducted between September, 1980 and February, 1981.

Two shallow, surface trenches are also reported to have been sampled and mapped sometime during the period 1979 to 1981. Shell reported non-compliant "drill indicated reserves" of 25.9 Mt and 30.0 Mt grading 0.21% tin in 1980, based on two different estimation methods using the results from 87 diamond drill holes completed in 1979 (Wilson, 1980).

The south-central part of the deposit was also tested by 975 m of underground drifting during the period from September, 1980 to February, 1981. The ramp access tested an area of approximately 500 m by 350 m to a vertical depth of 50 m. A total of 31,600 t of material was extracted as a bulk sample and four underground diamond drill holes totalling 490 m were drilled for comparative purposes. (RAL, 1983).

Detailed, deposit-scale and regional exploration for similar deposit styles was also carried out during the period 1979 to 1982 by Shell and several other major mining companies in the area. Exploration work and limited drilling by Shell also led to the discovery of several similar but smaller occurrences of greisen-style mineralization along the main sediment/granite contact for a distance of approximately 20 km, as well as a significant, sediment-hosted tin deposit at Duck Pond, which is located approximately 2 km to the northwest of the East Kemptville deposit.

Historic resources for Duck Pond have been reported at 9 Mt grading 0.11% Sn (Kooiman, 1989). This resource estimate is historic and does not satisfy the requirements set out in NI 43-101 and a Qualified Person has not completed sufficient work to classify the historic estimate. Therefore, the estimate may not be relied upon and should be regarded as conceptual in nature.

In 1982, the East Kemptville Deposit and surrounding claims were purchased from Shell by Riocanex, the Canadian exploration arm of RAL. During the years 1982 to 1983, RAL conducted a detailed due diligence of Shell's work, which included the resurveying of all possible surface diamond drill holes and re-assaying of several of Shell's drill holes. RAL also drilled a total of 15 HQ diameter holes totalling 1,305 m during 1983 in preparation for a feasibility study and production decision also completed in 1983.

Table 6.1 gives the historic preproduction geological and "mineable ore reserves" reported by RAL at the time of the feasibility study of 1983. These resources are shown solely for historic context and are not current and should not be relied upon.

Table 6.1
Historic Preproduction Geological and Mineable Ore Reserves (1983)

Category	Geological		Mineable					
	Tonnes	Sn (%)	Cut-off Grade Sn (%)	Tonnes	Sn (%)	Cu (%)	Zn (%)	Strip Ratio
Higher Grade Ore	40,193,951	0.194	0.12	40,800,000	0.185	0.11	0.18	1:1
Low Grade Ore	25,907,594	0.095	0.08	15,300,000	0.105			
Total	66,101,545	0.155		56,100,000	0.163			

Source: RAL, 1983.
Notes:
1. Resources are historic estimates. While viewed as reliable and relevant based on the information and methods used at the time, they do not satisfy the requirements set out by NI 43-101. Sampling and assay methods used may have resulted in understatement or overstatement of the grade and thus contained tin, copper and zinc. The extent of understatement or overstatement is unknown.
2. No Qualified Person has undertaken sufficient work to classify the historical estimates as current resources or reserves.
3. Resource classification not in accordance with CIM standards.
4. Avalon is not treating the historical estimate as current resources.
5. Historic resource estimate based on cut-off grade of 0.08% Sn for "low-grade ore" and 0.12% Sn for "higher-grade ore".
6. Estimate is a historic summary only. The Historical Resource should not be relied upon.

Open-pit operations at East Kemptville commenced in the fall of 1985 with a reported 17 years of production at a planned rate of 9,000 t/d of ore and 5,000 t/d of waste. Average annual concentrate production was expected to be 3,700 t of a high-grade tin concentrate grading 50% Sn; 8,600 t of low grade tin concentrate (21.4% Sn); 5,100 t of copper concentrate (25% Cu) and 4,000 t of zinc concentrate (50% Zn).

Shortly after commencing production, the operation ran into serious problems related to the recovery of tin by gravity methods. A dramatic price decline of approximately 50% for tin on world markets in the fall of 1985 put added fiscal pressure on the operation. Late in 1986, RAL announced the decision to end its financial obligations to the Project, and it was handed over to the lenders that had financed it (NSDNR Annual Report Summaries). The operation was re-purchased from the receiver by RAL and Rio Kemptville Tin Corporation took over control of the Project shortly thereafter.

A compilation of various sources of information in the public domain suggests that the operation was making headway in the last few years of operation with regard to improvements in tin recoveries and head grades. However, due to the continued depressed price of tin and the resultant low profit levels, operations were ceased in early 1992.

All annual production figures including low grade ore volumes, waste volumes and ore grade for the mine are not available; however, summary and partial grade/tonnage and concentrate figures have been compiled from a number of sources and give a reasonable estimate as to levels of total production and concentrates produced. Table 6.2 details total tonnage movements and annual mine production of ore (Sn% >0.14) during the operational life of the mine. This information has been confirmed by the following sources:

- East Kemptville Mine Closure Plan December, 1993.
- RAL annual reports 1985-1992.
- Rio Kemptville Tin Company 5-year Business Plan – 1991 (Dan Kontak, personal communication).

Table 6.2
Historic Mine Production (Dry Tonnes)

Rock or Ore Type	Life of Mine Production (1985-1992)
Overburden	650,000
Ore (>0.14% Sn head grade)	18,822,640
Ore (<0.14% Sn, >0.05% Sn)	5,870,000
Waste (<0.08% Sn)	3,963,000
Metasediments	3,013,000
Annual Ore Production (Dry Tonnes)	
1985	168,729
1986	2,703,034
1987	3,120,916
1988	3,213,358
1989	3,221,903
1990	3,175,122
1991	3,203,437
1992	16,141
Total	**18,822,640**

Source: RAL, 1993.

Compilation of data from the annual reports from the NSDNR put the total tin in concentrate production for the mine at 19,774 t; the total copper at 1,186 t; the zinc at 1,922 t and the silver production at 1,925 kg.

Diamond drilling in the immediate pit area during the life of the mine was limited to 42 BQ and 1 HQ diameter drill holes totalling 6,736 m. All of this drilling was completed by RAL during the period 1988 to 1991 primarily to target depth extensions to the mineralization.

From the period 1978 to 1991, both Shell and RAL conducted extensive exploration focused along the main contact between the granites and sediments, along the extension of the Baby Zone to the southwest, as well as more limited exploration work on the metasediment-hosted, Duck Pond deposit and Gardner's Meadow occurrence.

In 2006, Avalon acquired the mineral tenure to the mine site under a Special Licence 50462 which is currently in the process of renewal by the Government of Nova Scotia (Avalon has no reason to believe this will not be successfully completed). Subsequent compilation of historical exploration data, and limited field work where access was possible, by Avalon consultants over the last several years has indicated these areas are worthy of additional follow-up work. Well-mineralized, thick intercepts from relatively isolated drill holes located southwest of the Baby Zone at depth and in the Duck Pond area suggest further exploration work might be warranted in these areas.

In May 2014, Avalon entered into an agreement with the surface rights holder to secure access for an initial drilling program, and the agreement was extended to 2018 to allow for a second drilling program to be completed in the summer of 2015. A further extension to this agreement is currently being finalized

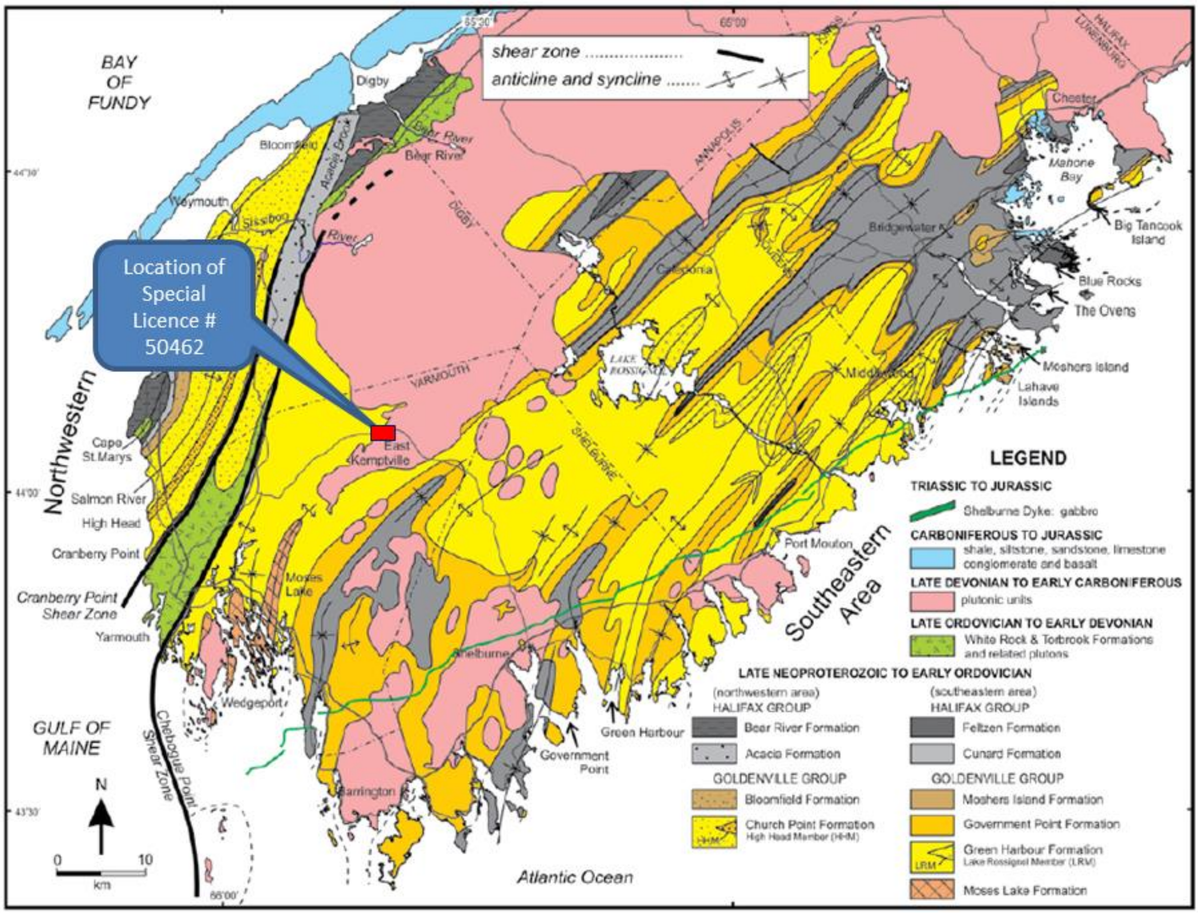
Avalon is currently assessing the economic viability of re-starting production on a relatively small scale, through the processing of a low-grade stockpile and high-grade zones of the Main and Baby Zone ore bodies (the subject of this report).

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The East Kemptville Project is located within the Cambro-Ordovician aged, Meguma Terrane of mainland Nova Scotia, a succession of Cambro-Ordovician-aged interbedded, metasedimentary rocks and intruded by Devonian-aged granites that comprise a large percentage of the bedrock geology of southern Nova Scotia (White, 2010) (see Figure 7.1).

Figure 7.1
Simplified Geological Map of Southwestern Nova Scotia



Source: White, 2010.

The Meguma Terrane has been subdivided into lower, sand-dominated facies, the Goldenville Formation, that is characterized by greywacke with minor slate or pelitic horizons and an upper, silt to clay-dominated pelitic facies, the Halifax Formation, that consists predominately of slate. The thickness of the Goldenville Formation is estimated at over 6,000 m and is conformably overlain by the Halifax Formation with an estimated thickness of approximately 4,500 m.

These strata were subjected to a regional deformation event, the Early to Middle Devonian Acadian Orogeny (ca. 410-380 Ma), associated with the docking or collision of the Meguma Terrane with the Avalon Terrane of northern Nova Scotia and Newfoundland. The rocks of the Meguma were subsequently folded into a series of long, northeast-trending, doubly plunging folds with anticlinal wavelengths in the order of 1 to 5 km, regionally metamorphosed from greenschist to amphibolite facies, and intruded by Middle to Late Devonian peraluminous granitoids (ca. 370-380 Ma) and minor mafic intrusions.

The South Mountain Batholith (SMB) represents the largest of these granitoid intrusions in western Nova Scotia and has been defined as a composite intrusion of older biotite granodiorite and smaller, younger leucocratic intrusions of Devonian-Carboniferous age. It is indicated on the geological map, Figure 7.1 as Late Devonian to Early Carboniferous Intrusives. Many of the granite-related tin occurrences and deposits in the province, including the East Kemptville tin deposit, appear to be related to the more evolved or specialized phases of the SMB.

Five deformation events (D1 to D5) have been identified that have affected the Meguma Group in Eastern Nova Scotia. These events are summarized by Ryan and Smith (1998) as follows:

> D1 – represented by early development of asymmetric step folds with subhorizontal grain alignment cleavage. These are variably developed and often difficult to determine due to later deformation.
>
> D2 – related to the pervasive regional folding event with upright slatey cleavage and pressure solution cleavage. This event is pre- to syn-regional metamorphism and is responsible for the geometry of the regional map patterns.
>
> D3 – coeval with 370 Ma granitoid plutonism and represented by rare folds and development of a flat lying crenulation cleavage or biotite foliation that are preferentially developed in the siltier lithologies.
>
> D4 – associated with ductile shear zones, small folds, upright cleavage and deformation of metamorphic porphyroblasts. Difficult to distinguish from fabrics developed during the D2 event.
>
> D5 – characterized by subhorizontal crenulation cleavage, thrust, normal, reverse and strike-slip faults, kink bands and boudinage structures. These fabrics generally appear to post-date gold mineralization, although exceptions to this have been noted at several gold districts where late stage kink structures are mineralized and where gold enrichment occurs along faults.

The structural fabric in southwestern Nova Scotia is characterized by a series of kilometre-scale, northeast-trending shear zones that appear to post-date the regional metamorphism. These shear zones are thought to have been partially to wholly re-set by thermal event such as the Alleghanian orogeny (ca. 300-295Ma) (Muecke, Elias, and Reynolds, 1988). In the East Kemptville area, a 1 to 4 km wide zone referred to as the East Kemptville Shear Zone (EKSZ) transects both the Meguma and the SMB granitoids. The EKSZ has been interpreted (Kontak and Cormier, 1991) to have been active from Devono-Carboniferous to Permian at circa 350-295 Ma (Kontak and Cormier, 1991) times.

The SMB consists of five distinct granitic phases, one of which has been referred to as the Davis Lake Complex (DLC) in the East Kemptville area. The DLC shows deformational fabrics characteristic of repeated movements along the EKSZ. Within the DLC itself there are distinct phases (i.e., biotite leucomonzogranite and muscovite bearing leucogranite) which may be the result of zonation and alteration within the pluton.

7.2 LOCAL AND PROPERTY GEOLOGY

The East Kemptville deposit is a greisen-hosted Sn-Cu-Zn-Ag-In deposit with the alteration and mineralization predominantly affecting the East Kemptville leucogranite (EKL). Three major lithologies (besides overburden) are recognized for the East Kemptville deposit. They are:

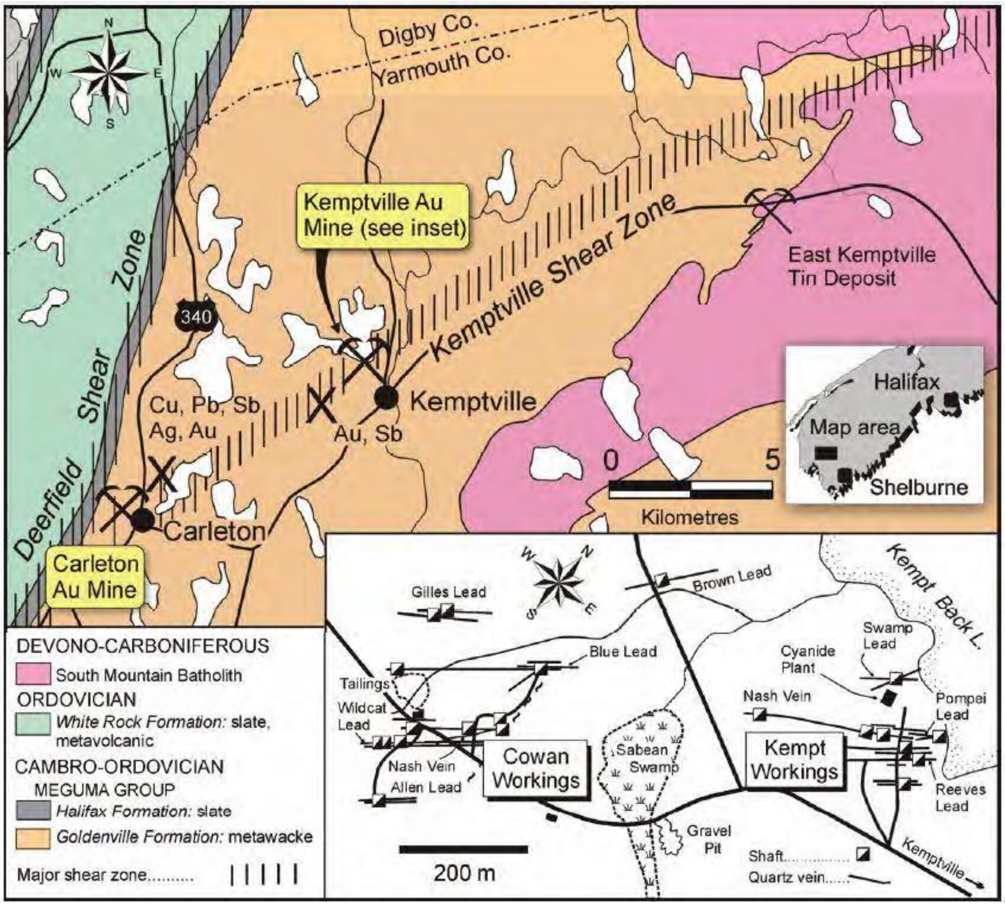
1. Leucogranite (EKL) intrusive which is primary host of the mineralization.
2. Within the Baby Zone, a contact zone which is a mineralized igneous breccia containing both metasediment and granite clasts.
3. Metasedimentary rock consisting of interbedded psammites and pelites.

The sub-vertical greisens generally strike at approximately 030°. It has been suggested that the geometry of the contact between the granite and the metasediment is the main control on grade and orientation of mineralization (RAL, 1983) or alternatively northeast trending faults, unrelated to the contact, may be the control (Bickerton, 2017, personal communication).

The East Kemptville deposit occurs on the southwestern edge of the DLC and is spatially associated with an apparent mushroom-shaped inflection, likely due to faulting (NW side upthrown/SE side downthrown), at the contact of an evolved, leucocratic phase of the DLC with the Meguma Group metasediments (See Figure 7.2). Geochemically, the topaz-bearing leucogranite that hosts the deposit is similar to other stanniferous granites globally and is enriched in fluorine, rubidium, tin and lithium (Kontak, 1990).

Tin and base metal (Zn-Cu-Ag-W) mineralization within the deposit is primarily fine to medium-grained and is associated with northeast-trending, sub-vertical and zoned, quartz-topaz, sulphide-bearing greisens, veins, and stockworks that occur primarily in the sericite-silica-topaz altered portions of the leucogranite near where the EKSZ (Figure 7.2) meet the roof zone in contact with surrounding metasediments. Mapping and studies of the deposit in the early 1990s suggest that structural controls related to the EKSZ may in fact be controlling some of the higher-grade mineralization at East Kemptville. Kontak (1990) suggests that the mineralization is syntectonic and that several of the sedimentary inliers within the deposit are structurally emplaced and control the higher-grade zones.

Figure 7.2
Local Geology, East Kemptville Area



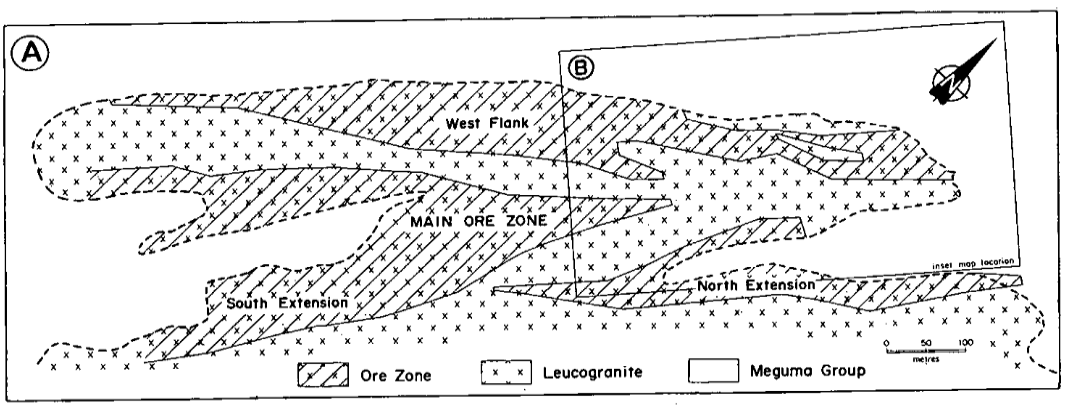
Source: O'Reilly, 2003.

The greisen mineralization is subdivided into 'zoned' and 'massive' mineralization types (Richardson, 1988; Halter et al., 1998). Massive greisens consists of large, elongate bodies (>20 m length) that exhibit quartz-topaz alteration and abundant, disseminated mineralization (Richardson, 1988). Zoned greisens are smaller, confined to individual veins, and typically exhibit a zoned alteration halo of quartz-topaz alteration near the vein with peripheral quartz-muscovite alteration. In both greisens, granite was altered by hydrothermal fluids to initially form a quartz-albite-muscovite greisen, and with further alteration forming quartz-muscovite and then quartz-topaz greisen. The quartz-topaz greisen alteration is accompanied by significant cassiterite and sulphide mineralization (Halter et al., 1998). Sulphide mineralization also occurs in the late, base-metal sulphide veins that cross-cut both zoned and massive greisens, and are found throughout the deposit (Kontak, 1994).

In many respects, the geometry of the mineralization over the areal extent of the deposit appears akin to a series of closely spaced jellyfish where the bodies represent wider zones of intense fracturing and coalescing greisens near the top of the intrusion and where the tentacles represent narrower, higher-grade feeders or root zones as described by Kooiman (1969) and other RAL geologists in the latter years of the previous mining operation. A structural control on the emplacement of the mineralization may help explain why some thicker (20-40 m) portions of the deposit occur at greater depths and why some thinner zones (2-5 m) occur at higher levels (i.e., northern portion of Main Pit). In general, the zones below the current pit appear to be thinning and are of higher grade, as noted from limited, deeper drilling completed on the deposit in 1989.

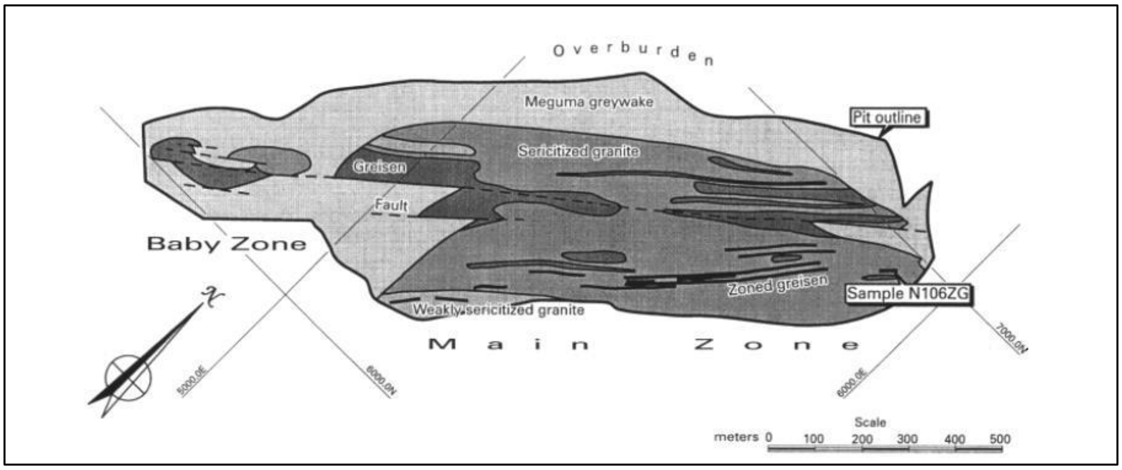
The overall gross dimensions of the original potential economic mineralization at the Main and Baby Zones based on a cut-off grade of approximately 0.05% Sn are in the order of 1,500 m long, 350 m wide and 75 m to 150 m deep. Most of this volume is represented by the larger, Main Zone, which has been further subdivided into three subzones termed the Western Flank and South and North Extension Zones (see Figure 7.3). The smaller and discrete Baby Zone occurs a few hundred metres southwest of the Main Zone within what is believed to be a structurally controlled, satellite intrusion (see Figure 7.4). The intrusion hosting the Baby Zone is in the order of 250 m long and 50 m to 75 m wide, but may be continuous with that of the Main Zone at depth through the South Extension Zone. There are on-going academic arguments regarding the presence of one or of more than one distinct phase of intrusive which may or may not be important for mineral exploration purposes.

Figure 7.3
Geological Plan of East Kemptville Deposit at 94-m Elevation



Note: This figure is historical and refers to "ore" zones. The terminology is not compliant with NI 43-101.
Source: Kontak and Dostal, 1992.

Figure 7.4
Geological Plan of the East Kemptville Deposit Showing Subzones



Source: Halter et al., 1996.

Higher grade, thicker, mineralization in the Baby Zone is focused along its structurally controlled, southeastern contact with the metasediments and along an interpreted fault that continues northeastward through the Main Zone (Kontak, 1990). Here, potentially economic mineralization reaches tens of metres in thickness and grades appear to average over 0.20% Sn, 0.30% Zn and 0.10% Cu. Narrower, higher-grade root zones have also been intercepted in deeper drilling beneath the Baby zone, as well as in the Main Zone. Bickerton (2017) stated that, "*the Main Zone is primarily hosted by the EKL and its highest grades are present along the EKSZ, whereas the higher-grade, lower tonnage Baby Zone is locally hosted by a magmatic-hydrothermal breccia and is possibly related to a different intrusive pulse, as indicated by lithogeochemical data*".

Bickerton (2017) has also noted the contrast in style between the Main and Baby Zones, explaining that the Main Zone mineralization is closely associated with ductile zones and greisens are predominantly restricted to fractures whereas the Baby Zone displays brittle deformation and has pervasive greisens below the wallrock contact.

Mineralization between the Main Zone and the Baby Pit is referred to as the Southwestern Extension of the Main Zone and is not exposed at surface but intersected in drilling.

7.3 MINERALIZATION

Two QEMSCAN® studies of selected samples from the Baby and Main Zones of the deposit were completed by SGS Lakefield Research/SGS Mineral Services (SGS) on behalf of Avalon in 2008 (SGS 2008) and 2009 (SGS, 2009). The work completed included:

- Modal abundance and grain size (BMA): 40 samples of the Baby Zone and 65 samples of the Main Zone (SGS, 2008).
- Liberation (PMA): 7 samples from Baby Zone and 4 samples from the Main Zone (SGS, 2008).

- Liberation (PMA): 5 samples from Baby Zone and 5 samples from Main Zone (SGS, 2009).
- Electron Microprobe (EMP) analyses: analysis of cassiterite, sphalerite and chalcopyrite grains from Main and Baby Zones (SGS, 2009).

Interpretation of the results of these studies has to take into account that the number of samples is limited and may not be totally representative of averages for the mineralized zones.

7.3.1 Modal Abundance and Grain Size

The samples consist of quartz, topaz, micas, chlorites, plagioclase, K-feldspar, and clays with trace amounts of zircon, other silicates, oxides, carbonates, fluorite and wolframite. Sulphide and other minerals include pyrite, copper sulphides, cassiterite, stannite, molybdenite, arsenopyrite, sphalerite, pyrrhotite and bismuthinite. Typical ranges for these minerals are as detailed in Table 7.1.

Table 7.2 and Figure 7.5 show the normalized percentage distribution of the grain size of cassiterite in the Main and Baby zones of the deposit. In general, cassiterite from the Main zone is coarser than that from the Baby zone, although both zones have a predominance of fine-grained cassiterite. The fine grain size of the cassiterite has significant implications for processing and recovery methods.

Table 7.1
Modal Abundance of Minerals (Percentage by volume)

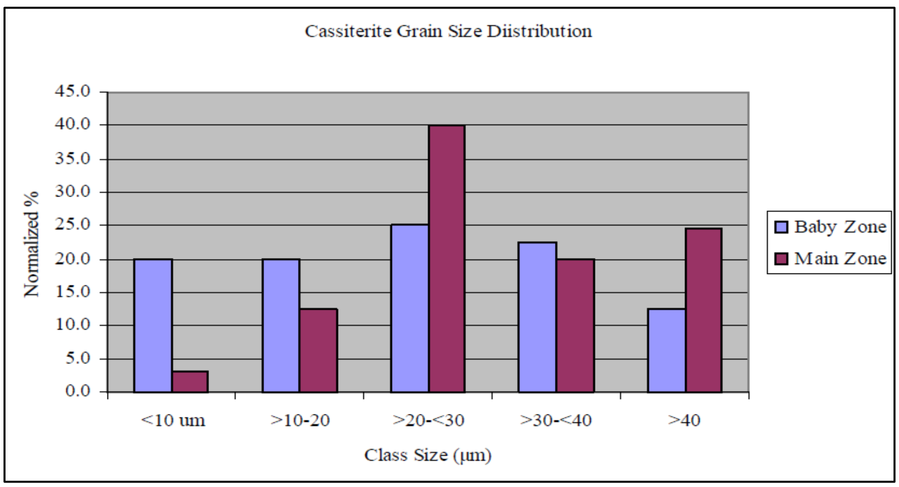
Mineral	Baby Zone			Main Zone		
	Minimum	Maximum	Average	Minimum	Maximum	Average
Pyrite	0.01	1.46	0.43	0.00	2.78	0.45
Cu Sulphides	0.00	0.93	0.20	0.01	2.51	0.35
Cassiterite	0.00	1.00	0.15	0.00	2.39	0.38
Stannite	0.00	0.07	0.01	0.00	0.24	0.02
Molybdenite	0.00	0.02	0.00	0.00	0.10	0.00
Arsenopyrite	0.00	0.32	0.01	0.00	0.08	0.00
Sphalerite	0.04	1.45	0.43	0.01	0.85	0.17
Pyrrhotite	0.03	2.33	0.97	0.00	3.08	0.35
Bismuthinite	0.00	0.09	0.01	0.00	0.16	0.02
Other Sulphides	0.00	0.03	0.00	0.00	0.01	0.00
Quartz	51.05	78.09	66.13	41.62	76.19	58.54
K_Feldspar	0.06	0.82	0.27	0.02	12.20	2.08
Plagioclase	0.02	8.90	1.06	0.06	14.98	5.48
Topaz	0.13	28.46	7.35	0.40	21.04	7.30
Micas	3.21	25.65	14.04	1.03	24.64	14.10
Clays	0.06	6.09	.79	0.07	6.59	2.61
Chlorites	4.17	13.86	7.00	1.27	22.59	6.63
Zircon	0.04	0.29	0.11	0.09	0.62	0.21
Other Silicates	0.01	0.16	0.06	0.00	0.08	0.02
Oxides	0.02	0.93	0.15	0.00	1.13	0.15

Mineral	Baby Zone			Main Zone		
	Minimum	Maximum	Average	Minimum	Maximum	Average
Carbonates	0.08	0.42	0.23	0.10	0.76	0.31
Fluorite	0.00	0.60	0.25	0.00	0.88	0.09
Wolframite	0.00	0.17	0.02	0.00	0.95	0.06
Other	0.14	0.82	0.34	0.38	1.55	0.69

Table 7.2
Cassiterite Grain Size Percent Distribution – Main and Baby Zones

Class Size	Baby Zone	Main Zone
<10 µm	20.0	3.1
>10-20	20.0	12.3
>20-<30	25.0	40.0
>30-<40	22.5	20.0
>40	12.5	24.6
Total	**100.0**	**100.0**

Figure 7.5
Grain Size Distribution of Cassiterite – Main and Baby Zones



7.3.2 Cassiterite Liberation

A comparison of cassiterite liberation based on particle mineralogical analysis (PMA) on minus 106-micron samples from the Main Zone and Baby Zone is shown in Figure 7.6. Liberation of cassiterite varies in both zones. A similar plot for the samples from the later (2009) study (Figure 7.7) shows a contrasting degree of liberation (SGS, 2009). The reasonsfor the contrasting results is that the 2009 data (Figure 7.7) shows the weight adjusted combined results of the four fractions: +106 microns, -106 + 53 microns, -53 + 20 microns and -20 microns.

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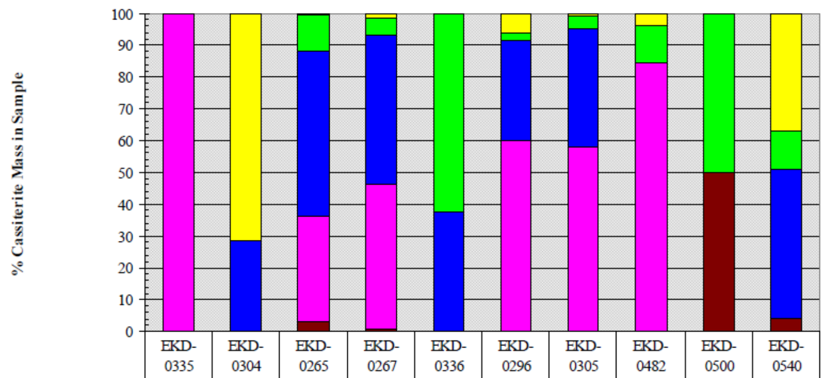
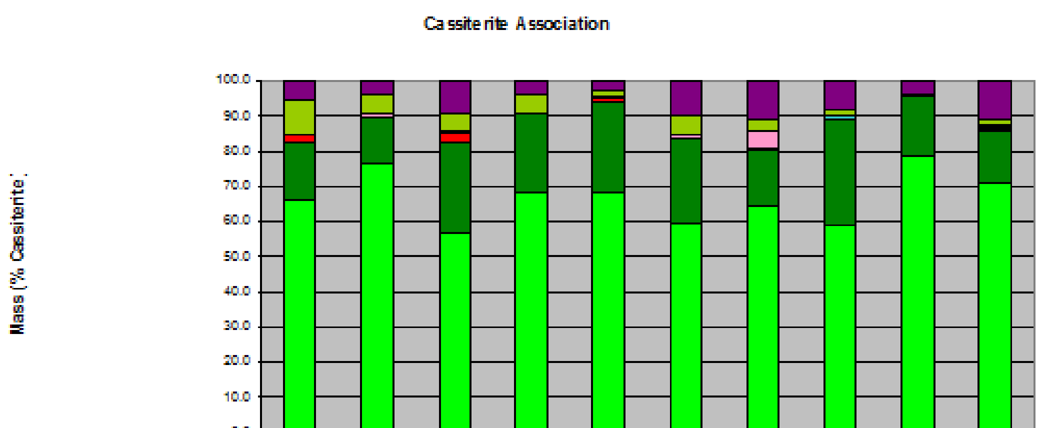
Figure 7.6
2008 Cassiterite Liberation (PMA Analysis) Main and Baby Zones



	EKD-0335	EKD-0304	EKD-0265	EKD-0267	EKD-0336	EKD-0296	EKD-0305	EKD-0482	EKD-0500	EKD-0540
Locked	0.0	71.4	0.3	1.4	0.0	6.1	0.9	3.8	0.0	36.7
Sub-middling	0.0	0.0	11.7	5.4	62.5	2.2	4.0	11.5	50.0	12.2
Middling	0.0	28.6	51.8	46.9	37.5	31.5	37.0	0.0	0.0	46.9
Liberated	100.0	0.0	32.9	45.6	0.0	60.1	58.1	84.6	0.0	0.0
Free	0.0	0.0	3.3	0.7	0.0	0.0	0.0	0.0	50.0	4.1

Note: Samples 335, 304, 265, 267, 336 and 296 from Baby Zone; 305, 482, 500, 540 from Main Zone.
Source: SGS, 2008.

Figure 7.7
2009 Cassiterite Liberation (PMA Analysis) Main and Baby Zones



	EKD-0217	EKD-0327	EKD-0004	EKD-0268	EKD-0295	EKD-0518	EKD-0489	EKD-0563	EKD-0622	EKD-0578
Complex	5.7	4.2	9.3	4.0	2.8	9.7	11.1	8.3	3.6	10.8
Cassiterite:Other	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0
Cassiterite:Micas/Chlorites/Clays	9.6	4.9	4.9	5.1	1.6	5.4	3.3	1.8	0.6	1.6
Cassiterite:Topaz	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.8	0.0	0.7
Cassiterite:Quartz	0.3	1.1	0.6	0.0	0.6	1.4	4.8	0.1	0.2	0.7
Cassiterite:Sphalerite	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0
Cassiterite:Cu Sulphides	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0
Cassiterite:Fe Sulphides	1.7	0.0	2.6	0.0	1.0	0.0	0.2	0.0	0.0	0.2
Lib Cassiterite	16.4	13.6	26.1	22.8	25.7	24.0	16.2	30.2	16.9	15.1
Free Cassiterite	66.4	76.1	56.5	68.1	68.3	59.3	64.3	58.8	78.6	70.9

Note: Samples 217, 327, 004, 268 and 295 from Baby Zone; 518, 489, 563, 622, 578 from Main Zone. Source: SGS, 2009.

Tin Deportment: Based on the modal abundance of the minerals and their tin content, it is shown that cassiterite accounts for most of the tin in the samples (average 92.73%), whereas stannite accounts for the remainder.

Zinc Deportment: Zinc is primarily found as sphalerite, with some enrichment in the Baby Zone compared to the Main Zone. The average modal abundance in the Baby Zone is 0.43% and that in the Main Zone 0.17%. It is noted that indium is associated with the sphalerite. Microprobe analyses by SGS (SGS, 2009) indicate an average indium content of 0.25% in sphalerite, but only 0.08% in chalcopyrite and below detection limit in cassiterite. More recent work by Jason Wilson (MSc Thesis, in progress) has reinforced these conclusions.

Copper Deportment: Copper is primarily present as chalcopyrite and a small portion in stannite. Other copper sulphide minerals such as bornite are present as trace quantities. The average modal abundance in the Baby Zone is reported as 0.20% while that for the Main Zone is reported as 0.35%.

7.3.3 Mineralogical Summary

Overall, the mineralogy of the deposit, as represented by the limited number of samples examined to date, can be summarized as follows (bearing in mind that these conclusions are reflecting information from a limited number of samples):

1. The mineralogy of the deposit consists mainly of similar contents of quartz, micas, feldspars, topaz and chlorites, although minor variations are noted in each group and between the Baby and Main Zones.
2. For the samples subjected to QEMSCAN® analysis, the average cassiterite content is higher in the Main Zone than in the Baby Zone (0.38% and 0.15% respectively), that of sphalerite is higher in the Baby Zone (0.43%) than that in the Main Zone (0.17%). The difference between these quantities and the actual metal grades in the deposit indicates that the QEMSCAN® samples are not totally representative of the contents in the Main and Baby Zones.
3. Cassiterite from the Main Zone is coarser than that from the Baby Zone. In the Baby Zone, approximately 40% of cassiterite is less than 20 µm, 47.5% is between 20 and 40 µm and only 12.5% is greater than 40 µm. In the Main Zone, the size distributions are 15.4%, 50% and 24.6%, respectively. Again, the small sample number should temper overall conclusions from this information.
4. The modal abundance and grain size of the minerals, as calculated using both the bulk mineral analysis (BMA) and PMA methods, are indicative of the limited variability of the minerals. However, analyses should be done on at least three or four size fractions for better determination of the modal abundance especially for the minor constituents in the samples, e.g., sphalerite, cassiterite.
5. Liberation and association of the minerals must be calculated based on size fractions and not on single samples/fractions.

6. Quartz seems to correlate well with plagioclase, potassium-feldspars and clays in the Main Zone, but poorly in the Baby Zone. Quartz and topaz show a generally poor correlation and partial overlapping. Copper sulphides versus sphalerite, cassiterite and topaz, as well as sphalerite versus cassiterite; also show a scatter, a partial overlap and a poor correlation. There is higher cassiterite content in the Main Zone and higher sphalerite in the Baby Zone. An overall positive correlation between quartz and the total amount of all sulphides is noted for both zones.

8.0 DEPOSIT TYPES

Tin deposit classification has been discussed by Taylor (1979) with reference to the classifications of Smirnov in Russia and Hosking in the United Kingdom. Taylor suggests a group of deposits termed "Deposits associated with passive and/or batholithic magmatic environments", and states that "this is the greisen association" or Erzgebirge style. This type of deposit is dominated in economic terms by massive greisen zones and quartz vein systems. It is suggested that this is the deposit type to which the East Kemptville deposit belongs.

9.0 EXPLORATION

Historical exploration on the Project is described in Section 6.0.

Prior to the 2014 and 2015 drill programs, exploration by Avalon has been limited to regional reconnaissance, geochemical sampling and limited diamond drilling on the exploration licences outside of the Special Licence area. The 2014 and 2015 drill programs are described in more detail in Section 10.0 of this report.

In 2009-2010, Avalon conducted a program of geochemical Mobile Metal Ion (MMI), induced polarization (IP) and gravity surveys on exploration Licences 07179, 08186 and 08720 in the Gardener's Meadows area approximately 6 to 7 km to the southwest of the East Kemptville mine area. The results of the work suggested the potential for a concealed granite cupola that may be responsible for the narrow, high-grade tin zones intersected from limited historical drilling and trenching. The geochemical and geophysical work identified an anomalous zone which was followed up by a six-hole NQ diamond drill program. Results of the drill program indicated the presence of several, narrow 0.10 m to 3.00 m zones of subeconomic polymetallic Sn-Zn-Cu-Ag-In grades associated with weakly developed zones of quartz-sulphide veining and alteration of the metasediments. No definitive evidence of a buried granite cupola was seen in the drill core.

Nevertheless, follow-up work consisting of in-fill drilling and additional multi-element geochemistry and IP geophysical work to identify possible vectors to wider, better mineralized zones of economic mineralization was recommended (Hudgtech, 2010).

The East Kemptville tin deposit is regarded as a brownfields exploration project. In certain respects, it can also be considered as an advanced exploration project. However, considerable work remains to be done to more fully define the economic potential of the remaining resources in the original deposit, the potential for delineating additional resources, and the costs associated with advancing the Project to a production decision.

9.1 EXPLORATION POTENTIAL

Exploration potential on the property is focused on following areas:

1. South and west of the Baby Pit, including the area referred to historically as the South Grid.
2. East of the Main Pit.
3. The Duck Pond area, north of the existing tailings ponds.

There are intercepts of tin mineralization reported in drill holes southwest of the Baby Zone in an area referred to in historic reports as the "South Grid Zone". This "zone" consists of two holes separated by 200 m with significant tin intercepts at a depth of about 90 m below the surface. Drill hole 90-010 has 34.3 m at 0.17% Sn and 90-008 has 33.0 m at 0.31% Sn, both in the granite. These intercepts are 1 km south along strike from the Main Pit mineralization and 700 m south of the Baby Pit mineralization. The mineralization is reportedly within granite.

The Main Zone appears to be open to the southeast and east. As a result, some of the 2015 drilling was in this area (see Section 10.0).

The Duck Pond deposit, located approximately 2 km west of the East Kemptville mine site, was drilled by Shell in 1980 (nine holes) and RAL in 1983/84 (seven holes). Avalon completed 5 drill holes for 1,153 m as part of the 2015 drill program on the property and described in more detail in Section 10.0.

The rocks in the Duck Pond deposit are interbedded argillites and greywackes in a transition unit from the Goldenville Formation to the Halifax Formation. The argillites are reported as fine-grained, chloritized and slatey. The coarser grained sandy argillites and silicified sediments or greywackes are sericite-muscovite rich. Tin mineralization associated with chloritic alteration, occurs in veinlets and as disseminated material in the silicified zones. Sphalerite, chalcopyrite and bornite are also present. Mineralization is reportedly present in two zones over a strike length of 750 m, open to the northeast, southwest and at depth. The Duck Pond Zone is underlain by a strong airborne magnetic anomaly, partially coincident with and extending beyond the known mineralization. It is possible that this magnetic anomaly is caused by an intrusive at depth which was the source of the mineralizing fluids. This suggests potential for tin-zinc mineralization at depth.

10.0 DRILLING

Historic drilling on the property by Shell and RAL is described in Section 6.0.

10.1 2014 AVALON DRILLING PROGRAM

The purpose of Avalon's 2014 drill program was to investigate mineralization between the Main Zone Pit and Baby Pit, referred to as the Southwestern Extension of the Main Zone, and at depth and to twin some selected historic holes as part of a due diligence program to validate the historical drill results. The drill program confirmed mineralization outlined by historical drilling.

Avalon completed an in-fill/twin hole program consisting of seven HQ diamond drill holes totalling 986 m in 2014, as summarized in Table 10.1. Four of the holes twinned historical drill holes while two targeted higher-grade mineralization underneath the Baby Pit. One drill hole targeted higher-grade mineralization between the Main and Baby Pits.

The seven HQ diamond drill holes were completed on Special Licence 1/12 by Logan Drilling Group International from July to September, 2014 in order to confirm tin and polymetallic zinc-copper mineralization. Core recovery for the seven holes was close to 100%.

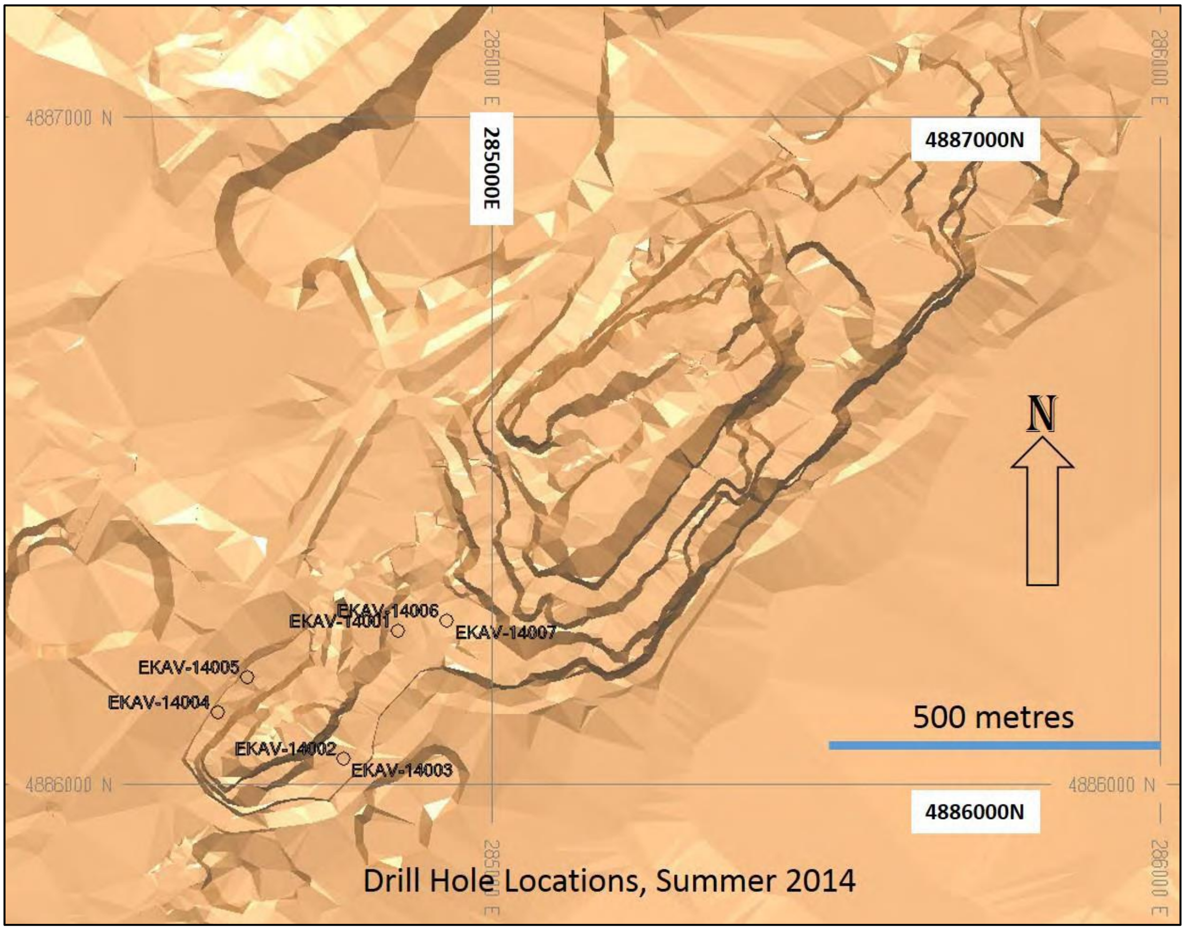
Holes were spotted using a handheld Garmin GPSmap 60Cx and, upon completion, were cemented into bedrock from top to bottom where possible. Downhole surveys were performed on all holes using the Reflex system. Coordinates in NAD27 and NAD83/Zone 20 for each hole were established upon completion by Scotia Surveys Limited of Shelburne, Nova Scotia, using a Topcon HiperLitr +GPS RTK system. All drill sites were cleaned upon completion. The core is presently stored on the property at 3762 Highway 203, East Kemptville which is owned by Avalon.

Table 10.1
Details of Drill Holes – 2014 Drill Program

Drill Hole	Field Data					Dates		
	Easting	Northing	Dip	Azimuth	Length	Start date	End date	Days
EKAV-14-001	284,911.0	4,886,454.0	-45	120	147	22-Jul-14	01-Aug-14	10
EKAV-14-002	284,831.0	4,886,264.0	-65	300	159	02-Aug-14	12-Aug-14	10
EKAV-14-003	284,830.5	4,886,264.5	-45	300	148.1	13-Aug-14	20-Aug-14	7
EKAV-14-004	284,638.0	4,886,337.0	-45	120	166	21-Aug-14	30-Aug-14	9
EKAV-14-005	284,684.0	4,886,338.0	-45	120	145	31-Aug-14	03-Sep-14	3
EKAV-14-006	284,982.0	4,886,472.0	-83	120	113	05-Sep-14	09-Sep-14	4
EKAV-14-007	284,982.0	4,886,472.0	-45	120	106	09-Sep-14	12-Sep-14	3

Locations of drill holes in this program are shown in Figure 10.1.

Figure 10.1
Drill Plan, Summer 2014



Note: Topography is the 1992 post mining surface. Source: Avalon.

10.1.1 2014 Drill Program Results

In general, the geology and polymetallic Sn + Zn + Cu zones encountered in the drilling are considered to be typical of historic drill results reported by Shell and RAL in the Baby and Southwest Extension Zone Areas.

Lithologies intersected in the holes consist predominately of Meguma metasedimentary rocks and felsic, granitic intrusives. The metasedimentary rocks are light to steel grey, weakly to moderately siliceous greywacke that contain minor, narrow interbeds of grey green argillite (commonly less than a metre). Normal Meguma turbidity sequences are typical along with common millimetre-scale micro fracture-filling quartz veining which may be mineralized. Weakly hornfelsed units are not uncommon and appear more frequently closer to the main granite intrusion.

Significant drill intercepts are summarized in Table 10.2.

Table 10.2
Summary Drill Hole Intercepts, 2014 Drill Program

Zone	Drill Hole	From (m)	To (m)	Width (m)	Zn (ppm)	Sn (ppm)	Cu (ppm)	Ag (ppm)	In (ppm)
Southwest Extension	EKAV-14-001	77.90	109.00	31.10	1,179	1,357	507	1.18	7.19
	including	77.90	98.50	20.60	1,104	1,791	659	1.52	7.18
Southwest Extension	EKAV-14-006	29.70	110.00	80.30	2,270	2,623	1,142	3.41	14.92
	including	29.70	83.00	53.30	2,350	3,270	1,412	4.31	15.59
	and	95.00	110.00	15.00	3,204	2,229	820	2.17	20.49
Southwest Extension	EKAV-14-007	44.50	82.00	37.50	1,828	1,567	617	1.70	11.67
	including	44.50	73.00	28.50	2,078	1,883	607	1.77	12.45
	and	44.50	48.80	4.30	3,177	1,555	1,154	3.26	26.03
	and	55.00	64.00	9.00	2,999	4,156	773	2.32	15.03
Baby Zone	EKAV-14-002	92.25	159.00	66.75	5,039	3,599	949	2.65	20.76
	including	92.25	133.50	41.25	3,924	4,691	1,225	3.36	21.00
Baby Zone	EKAV-14-003	49.00	65.75	16.75	2,890	3,897	801	1.80	20.91
	and	80.25	148.10	67.85	5,349	4,101	482	1.01	16.77
Baby Zone	EKAV-14-004	63.65	166.00	102.35	4,365	942	288	0.53	13.78
	including	63.65	161.20	97.55	4,521	975	293	0.54	14.20
	including	63.65	101.50	37.85	4,805	1,141	376	0.48	14.72
	and	80.50	94.00	13.50	5,218	1,702	489	0.68	16.26
	and	128.50	134.50	6.00	1,960	1,355	69	0.25	7.11
	and	147.00	161.20	14.20	11,524	1,685	583	1.08	30.19
Baby Zone	EKAV-14-005	No significant intercepts							

The main granite intrusion is weakly to strongly altered with alteration affecting both the colour and texture of the granite. Generally, light to darker, highly siliceous, fine grained granite is associated with the higher grade, mineralized sections. Yellow to yellowish grey, medium to coarse grained granite with abundant sericite is most commonly associated with lower grade mineralization adjacent to the higher-grade mineralization.

The unit associated with the lower grade mineralization has a gradational contact with unaltered and unmineralized granite.

The following types of veining occur:

1. Quartz veining as bedding-parallel veins, angular veins and irregular fracture filling millimetre-scale veins. These veins are commonly barren but sometimes have minor to trace mineralization consisting of chalcopyrite, + sphalerite, cassiterite, pyrrhotite, arsenopyrite and possibly molybdenite.
2. Quartz flooding or replacement style veining/silification (greisenization) which commonly has associated mineralization (+ sphalerite, + cassiterite + pyrrhotite, + pyrite, + fluorite, + molybdenite, + wolframite).
3. Semi-massive to massive sulphide veins, consisting predominately of pyrrhotite, + chalcopyrite and sphalerite. Cassiterite is locally abundant in these veins.
4. Cream coloured, fine-grained, granitic veins or dykes commonly associated with high-grade tin mineralization.

All drill holes were logged in detail, noting lithology, alteration and mineralogy. Density, rock quality designation (RQD) and magnetic susceptibility readings were also recorded for all drill holes.

10.1.2 Drill Hole Summaries

EKAV-14-001: Diamond drill hole EKAV-14-001 was drilled to twin historical drill holes 79-047 and 89-208 (extension of 79-047). The geology intersected is summarized as follows: 0 to 4 m overburden, 4 m to 77.9 m metasediments and 77.9 m to end of hole at 147 m is granite. The geology is consistent with the historical drilling and from examinations of the analytical data, a similar trend in grade is recognized.

EKAV-14-002: Diamond drill hole EKAV-14-002 was drilled to test-higher grade, thicker, mineralization in the Baby Zone that is focused along the southeast, structurally controlled contact with the metasediments. The geology intersected is summarized as follows: 0 to 3 m overburden, 46.6 m to 97.95 m contact zone (Igneous Breccia) and granite from 97.95 m to the end of the hole at 159 m. The drilling cut a thicker zone of contact zone geology than anticipated. This appears largely due to drilling sub-parallel to the zone, however, the zone is noted to be thicker in this local area as compared to other parts of the deposit. Drilling confirmed the higher grade, thicker zone of mineralization focused along the southeast contact.

EKAV-14-003: Hole EKAV-14-003 was drilled from the same set up as EKAV-14-002 to test the higher grade, thicker, mineralization in the Baby Zone that is focused along the southeast, structurally controlled contact with the metasediments at a shallower depth. The geology intersected is summarized as follows: 0 m to 3.5 m overburden, 3.5 m to 32.35 m metasediments, 32.35 m to 49 m contact zone, 49 m to 65.75 m granite, 65.75 m to 81.25 m contact zone and from 81.25 m to the end of hole at 148.1 m is granite. Within the main granite, hole EKAV-14-003 intersected an inlier of contact zone that was not recognized from the historical drilling. The significance of this inlier is not well understood at this time and it is recommended that the zone should be completely sampled and analysed.

EKAV-14-004: Hole EKAV-14-004 was drilled to twin historical drill hole 88-200 and to test the higher grade, thicker, mineralization in the Baby Zone that is focused along the southeast structurally-controlled contact with the metasediments. The geology intersected is summarized as follows: 0 to 3 m overburden, 3 m to 63.65 m metasediments, 63.65 m to 65.7 m contact zone, 65.7 m to- 161.2 m granite and from 161.2 m to the end of the hole at 166 m is contact zone. The geology is consistent with the historical drilling and from examinations of the analytical data, a similar trend in grade is recognized.

EKAV-14-005: Hole EKAV-14-005 was drilled to twin historical drill hole 90-003. The geology is 0 to 10.7 m overburden, 10.7 m to 63.1 m metasediments and from 63.1 m to the end of the hole at 145 m is granite. The geology is consistent with the historical drilling. However, no significant mineralization was encountered in this hole.

EKAV-14-006: Hole EKAV-14-006 was drilled to test the higher-grade mineralization noted at the end of historical drill hole 80-160. The geology intersected is as follows: 0 to 2.7 m overburden, 2.7 m to 28.35 m is metasediments and from 28.35 m to the end of the hole at 113 m is granite. The thickness of the metasediments is greater than anticipated from the historical drilling. In addition, this hole confirmed significant mineralization that occurs past the end of historical drill hole 80-160.

EKAV-14-007: Hole EKAV-14-007 was drilled to twin historical drill hole 80-159. The geology comprises: 0 to 5 m overburden, 5 m to 40.65 m metasediments and from 40.65 m to end of the hole at 106 m is granite. Preliminary observations indicate the geology and grade are consistent with historical drilling results.

10.2 CONCLUSIONS

The main conclusions from the 2014 drill program were:

- Drilling was successful in confirming the known geology.
- Drilling confirmed higher-grade, thicker mineralization focused along the southeast, structurally controlled contact of the Baby Zone with the metasediments.
- Diamond drill hole EKAV-14-003 identified a 'contact zone-like' inlier of sediments with the main granite that was not previously recognized.
- No significant difference in grade of mineralization or geological interpretations were noted or recognized from this preliminary evaluation.
- Drilling confirmed the mineralization associated with the southwest extension of the Main Zone.

10.3 2015 AVALON DRILLING PROGRAM

Twenty-two HQ diamond drill holes totalling 4,514 m were completed on Special Licence No. 50462 by Foraco Drilling from 13 July to 15 November, 2015. All drilling was performed on RAL and Crown Lands.

Holes were spotted using a handheld Garmin GPSmap 60Cx and, upon completion, were cemented into bedrock from top to bottom where possible. In cases where drill holes were not totally cemented, a plug was inserted at about 30 m depth in the hole and the hole was cemented from there to surface.

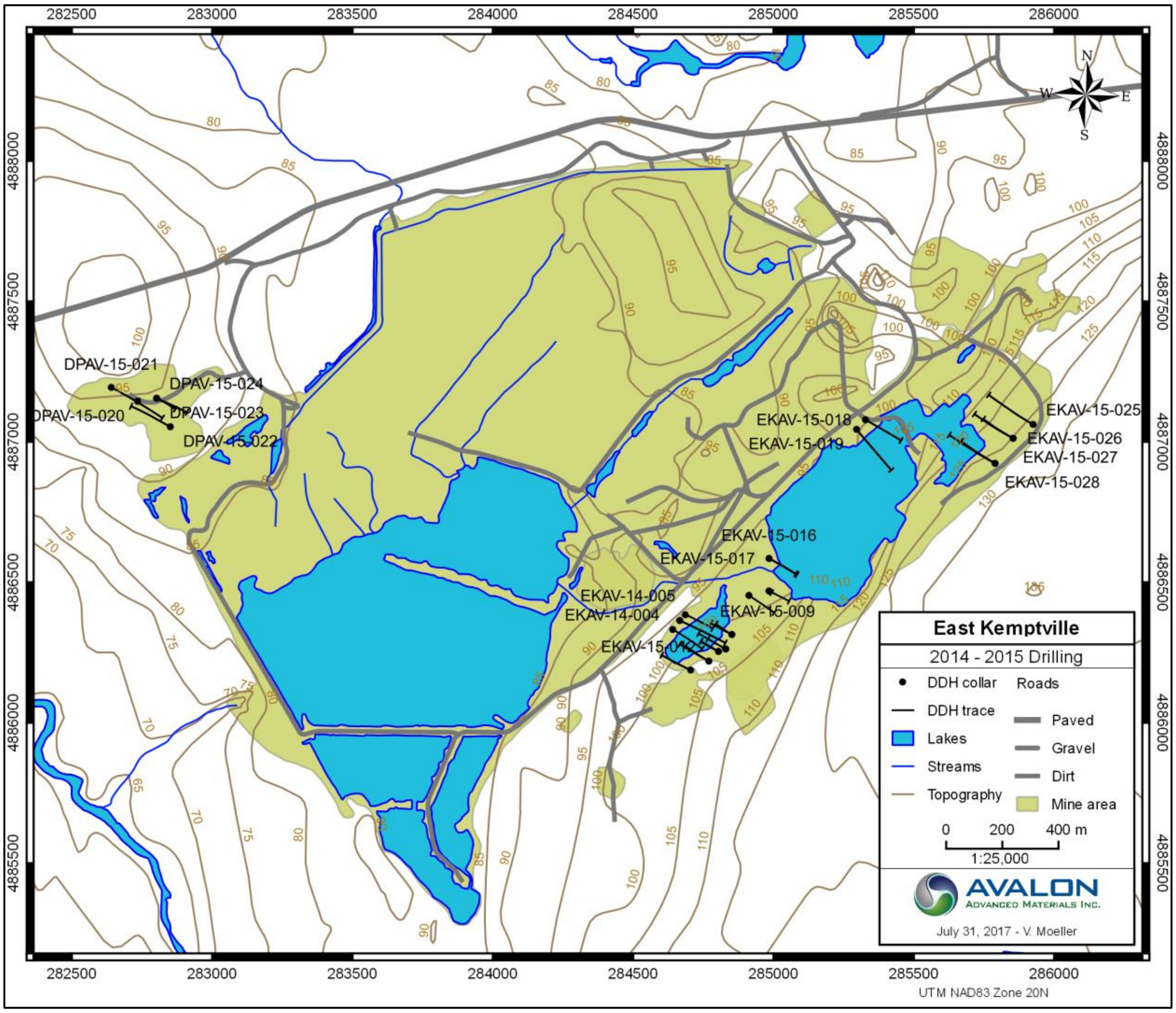
Surveyed coordinates for each hole were established upon completion by Scotia Surveys Limited of Shelburne, N.S. using a Topcon HiperLitr +GPS RTK system. All drill sites were cleaned upon completion. The core is presently stored at 3193 Main Shore Road, Port Maitland, N.S.

Completed drill holes are tabulated below in Table 10.3 and a diagram of the 2015 drill plan is presented in Figure 10.2.

Table 10.3
Drill Hole Details

DDH	E (NAD83)	N (NAD83)	Elevation (m)	Dip (degrees from horiz)	Azimuth (rel to true north)	START	FINISH	Planned (m)	EOH (m)	CUMUL. (m)
EKAV-15-008	4886314	284852	79	-70	300	13-Jul-15	16-Jul-15	150	173.5	173.5
EKAV-15-009	4886314	284851	79	-60	300	17-Jul-15	20-Jul-15	150	164.5	338
EKAV-15-010	4886254	284804	80	-70	300	21-Jul-15	06-Aug-15	200	191.5	529.5
EKAV-15-011	4886254	284804	80	-55	300	06-Aug-15	08-Aug-15	90	122	651.5
EKAV-15-012	4886220	284770	81	-60	300	09-Aug-15	12-Aug-15	200	185	836.5
EKAV-15-013	4886220	284769	81	-45	300	12-Aug-15	14-Aug-15	105	161	997.5
EKAV-15-014	4886188	284705	79	-45	300	15-Aug-15	18-Aug-15	115	155	1,152.50
EKAV-15-015	4886364	284666	73	-45	120	19-Aug-15	23-Aug-15	200	251	1,403.50
EKAV-15-016	4886585	284984	72	-50	122	01-Sep-15	04-Sep-15	150	164	1,567.50
EKAV-15-017	4886585	284984	72	-40	122	05-Sep-15	08-Sep-15	125	144	1,711.50
EKAV-15-018	4887079	285328	74	-40	122	09-Sep-15	14-Sep-15	175	182	1,893.50
EKAV-15-019	4887045	285296	77	-45	143	15-Sep-15	20-Sep-15	250	257	2,150.50
DPAV-15-020	4887144	282734	69	-90	0	30-Sep-15	03-Oct-15	250	260	2,410.50
DPAV-15-021	4887194	282640	76	-45	120	04-Oct-15	07-Oct-15	250	275	2,685.50
DPAV-15-022	4887054	282851	68	-45	300	09-Oct-15	13-Oct-15	250	224	2,909.50
DPAV-15-023	4887155	282802	68	-45	120	14-Oct-15	16-Oct-15	150	167	3,076.50
DPAV-15-024	4887156	282802	68	-70	120	16-Oct-15	19-Oct-15	175	227	3,303.50
EKAV-15-025	4887062	285925	102	-45	300	29-Oct-15	03-Nov-15	250	245	3,548.50
EKAV-15-026	4887013	285854	100	-60	300	04-Nov-15	06-Nov-15	250	230	3,778.50
EKAV-15-027	4887013	285853	100	-45	300	07-Nov-15	09-Nov-15	200	218	3,996.50
EKAV-15-028	4886924	285789	104	-45	300	10-Nov-15	12-Nov-15	225	248	4,244.50
EKAV-15-029	4886924	285790	104	60	300	13-Nov-15	15-Nov-15	250	269	4,513.50

Figure 10.2
2015 Drill Plan



10.3.1 2015 Drill Program Results

Lithologies intersected in the holes consist predominately of Meguma metasedimentary rocks and felsic, granitic intrusives. The metasedimentary rocks are light to steel grey, weakly to moderately siliceous greywacke that contain minor, narrow interbeds of grey green argillite (commonly less than a metre). Normal Meguma turbidity sequences are typical along with common millimetre-scale micro fracture-filling quartz veining which may be mineralized. Weak hornfels units are not uncommon and appear more frequently closer to the main granite intrusion.

The main granite intrusion is weakly to strongly altered with alteration affecting both the colour and texture of the granite. Generally, light to darker, highly siliceous, fine grained granite is associated with the higher grade, mineralized sections. Yellow to yellowish grey, medium to coarse grained granite with abundant sericite is most commonly associated with lower grade mineralization adjacent to the higher-grade mineralization.

The following types of veining were noted in the drilling:

1. Quartz veining as bedding-parallel veins, angular veins and irregular fracture filling millimetre-scale veins. These veins are commonly barren but sometimes have minor to trace mineralization consisting of chalcopyrite, + sphalerite, cassiterite, pyrrhotite, arsenopyrite and possibly molybdenite.
2. Quartz flooding or replacement style veining/silification (greisenization) which commonly has associated mineralization (+ sphalerite, + cassiterite + pyrrhotite, + pyrite, + fluorite, + molybdenite, + wolframite).
3. Semi-massive to massive sulphide veins, consisting predominately of pyrrhotite, + chalcopyrite and sphalerite. Cassiterite is locally abundant in these veins. Additional readings were also taken for density, rock quality designation (RQD) and magnetic susceptibility.

The 22 drill holes totaling 4,514 m achieved the following objectives, namely:

- Further definition of mineral resources.
- Geotechnical information for future mine planning.
- Geochemical information for waste rock handling planning.
- Potential bulk sample for pilot scale metallurgical testing.

The drill program was successful in providing assay results that are the basis of the update of the mineral resources for the East Kemptville Project, which includes three zones:

- Baby Zone.
- Duck Pond Zone.
- Main Zone.

The drill hole sample preparation and assays, conducted under best practice QA/QC procedures with insertion of blanks and standards, as well as duplicate coarse sample analyses at a secondary laboratory and core duplicates, indicate similar values of economic elements such as tin and zinc as historic data, allowing verification of the data by an independent Qualified Person.

Basic geotechnical information was collected by logging of all holes with a limited number representative holes logged in greater detail. In addition, wall rock samples (rock material outside, but adjacent to the mineralization zones) were analysed for carbon and sulphur in order to provide information on potential for acid rock drainage in the waste stripping process.

Drill core rejects are retained for metallurgical testing purposes enabling the possibility of a pilot plant based on 2015 drill core rejects. It is estimated that the drill core rejects would provide at least 3 tonnes of sample for a pilot plant test.

Significant drill intercepts are summarized in Table 10.4 to Table 10.7.

Table 10.4
Drill Intercepts Baby Zone – 2015 Drilling

From	To	Width	Sn %	Zn %	Cu %
EKAV-15-08					
No significant Sn intercepts					
EKAV-15-09					
74.3	110.5	36.3	0.23	0.33	0.14
EKAV-15-10					
76.0	158.3	82.3	0.36	0.63	0.07
76.0	140.3	64.3	0.42	0.71	0.08
EKAV-15-11					
68.4	71.3	2.8	0.52	0.69	0.05
85.3	122.0	36.7	0.15	0.69	0.06
85.3	104.0	18.7	0.22	0.64	0.08
EKAV-15-12					
Zn values but no significant Sn intercepts					
102.0	185.0	83.0	0.05	0.24	0.02
EKAV-15-13					
89.0	156.5	67.5	0.10	0.39	0.03
89.0	99.5	10.5	0.18	0.68	0.05
EKAV-15-14					
No significant Sn intercepts					

Table 10.5
Drill Intercepts Main Zone West – 2015 Drilling

From	To	Width	Sn %	Zn %	Cu %
EKAV-15-15					
81.5	83.0	1.5	1.00	2.05	0.05
175.5	188.0	12.5	0.14	0.33	0.05
238.6	248.0	9.4	0.40	0.35	0.04
EKAV-15-16					
45.5	47.0	1.5	0.76	0.05	0.07
57.5	59.0	1.5	1.00	0.02	0.16
69.5	89.0	19.5	0.21	0.22	0.12
137.0	143.0	6.0	0.08	0.02	0.32
156.5	158.0	1.5	0.72	0.03	0.21
EKAV-15-17					
24.2	135.0	110.8	0.12	0.09	0.09
24.2	33.5	9.3	0.23	0.10	0.18
44.0	52.7	8.7	0.21	0.03	0.07
75.6	93.5	17.9	0.26	0.27	0.17
111.5	119.4	7.9	0.22	0.25	0.25
129.7	135.0	5.3	0.43	0.07	0.31

From	To	Width	Sn %	Zn %	Cu %
EKAV-15-18					
99.5	111.5	12.0	0.18	0.38	0.15
122.0	127.5	5.5	0.21	0.30	0.03
155.0	158.5	3.5	0.54	0.13	0.13
EKAV-15-19					
106.8	203.0	96.2	0.13	0.10	0.08
106.8	116.0	9.2	0.33	0.30	0.15
128.2	135.8	7.6	0.14	0.26	0.08
145.0	146.3	1.3	1.00	0.12	0.09
159.5	168.3	8.8	0.26	0.12	0.33
179.4	188.1	8.7	0.25	0.05	0.14
191.1	203.0	11.9	0.12	0.04	0.05

Table 10.6
Drill Intercepts Main Zone East – 2015 Drilling

From (m)	To (m)	Width (m)	Sn %	Zn %	Cu %	In ppm
EKAV-15-025						
4.5	59.6	55.1	0.10	0.18	0.02	5.5
INCLUDING						
47.5	51.5	4.0	0.71	0.13	0.01	5.3
81.3	92.0	10.7	0.22	0.37	0.05	21.7
102.0	104.1	2.1	0.15	0.32	0.06	20.5
108.7	129.5	20.8	0.38	0.56	0.14	32.8
134.0	144.5	10.5	0.21	0.18	0.04	10.4
162.5	167.0	4.5	0.34	0.10	0.02	3.9
176.0	179.0	3.0	0.15	0.05	0.02	3.2
189.5	212.0	22.5	0.07	0.26	0.05	8.3
215.2	225.5	10.4	0.27	0.42	0.10	17.7
243.5	245.0	1.5	0.36	0.07	0.02	1.9
EKAV-15-026						
5.0	6.5	1.5	0.45	0.59	0.10	12.2
38.0	47.9	9.9	0.38	0.49	0.11	21.9
66.4	84.5	18.1	0.19	0.27	0.06	12.1
89.0	95.0	6.0	0.13	0.12	0.02	5.3
114.4	116.0	1.7	2.76	0.30	0.19	28.2
119.0	137.2	18.2	0.13	0.19	0.07	13.4
158.0	163.1	5.1	0.31	0.14	0.08	9.7
192.5	198.1	5.6	1.30	0.20	0.64	40.0
201.6	206.0	4.4	0.16	0.35	0.12	12.4
EKAV-15-027						
20.0	26.0	6.0	0.20	0.19	0.03	4.8
53.0	62.0	9.0	0.08	0.31	0.02	9.2
66.0	70.7	4.7	0.59	0.48	0.04	14.4
78.5	85.0	6.5	0.31	0.28	0.09	14.8
94.8	101.2	6.4	0.37	0.38	0.21	24.7
105.5	117.5	12.0	0.14	0.12	0.04	6.9
168.7	178.0	9.3	0.33	0.77	0.26	39.9
207.0	213.5	6.5	0.14	0.07	0.02	4.3
EKAV-15-028						
48.5	50.0	1.5	0.46	0.20	0.02	7.6

From (m)	To (m)	Width (m)	Sn %	Zn %	Cu %	In ppm
72.5	87.5	15.0	0.19	0.25	0.09	12.0
110.0	111.5	1.5	0.43	0.07	0.01	4.1
137.0	138.5	1.5	0.48	0.08	0.03	4.2
144.5	154.1	9.6	0.34	0.34	0.25	27.9
159.5	162.2	2.7	0.13	0.20	0.03	7.4
180.5	182.0	1.5	0.48	0.29	0.13	14.1
204.0	209.0	5.0	0.29	0.13	0.07	6.1
EKAV-15-029						
27.5	30.5	3.0	0.30	0.20	0.05	10.6
44.1	45.8	1.7	1.27	0.44	0.02	20.3
56.0	58.0	2.0	0.05	0.23	0.03	7.9
126.5	132.1	5.6	0.31	0.11	0.09	0.0
165.3	168.3	3.0	0.16	0.23	0.35	26.6
191.0	194.0	3.0	0.13	0.15	0.08	11.0
221.0	222.5	1.5	0.40	0.25	0.04	12.9

Table 10.7
Drill Intercepts Duck Pond Zone – 2015 Drilling

From (m)	To (m)	Width (m)	Sn %	Zn %	In ppm
DPAV-15-20					
96.5	98.0	1.5	0.79	0.20	7.2
129.4	156.4	27.0	0.23	0.39	8.4
216.5	236.0	19.5	0.43	0.25	12.1
257.0	260.0	3.0	0.41	1.24	36.0
DPAV-15-021					
Significant intercepts are zinc with low tin					
DPAV-15-022					
14.0	21.5	7.5	0.86	0.20	37.6
30.3	35.2	4.9	0.15	0.21	11.6
43.2	56.0	12.8	0.16	0.13	21.2
DPAV-15-023					
40.5	43.0	2.5	0.39	0.18	14.0
89.5	120.5	31.0	0.15	0.25	9.6
DPAV-15-024					
6.5	9.0	2.5	0.16	0.05	
50.5	61.0	10.5	0.60	0.27	18.0
135.5	143.1	7.6	0.28	0.38	6.0

10.3.2 Drill Hole Summaries

Abbreviations used in the drill hole summaries are listed in Table 10.8.

Table 10.8
Abbreviations Used in Drill Hole Summaries

Abbreviation	Mineral Name
aspy	Arsenopyrite
cas	Cassiterite
cpy	Chalcopyrite
flu	Fluorite
ga	Galena
moly	Molybdenite
po	Pyrrhotite
py	Pyrite
qtz	Quartz
sph	Sphalerite
wolf	Wolframite

Baby Zone: The initial ten drill holes were all targeted on the Baby Zone and Table 10.9 below summarizes the results.

Table 10.9
Summary of Initial 10 Holes Drilled in the Baby Zone

DDH	Objective	Comments Mineralization	Comments: Water, Etc.	Comments: Cementing
EKAV-15-008	Baby Zone: Upgrading resources	Meguma better mineralized than granite. Trace mineralization upper granite.	Casing leaking cuttings. No water loss in hole although return occasionally diminished.	24 eighty-pound bags used.
EKAV-15-009	Baby Zone: Upgrading resources	Transition Zone moderately mineralized and upper part of granite to 110.5 m. Minor mineralization to 145 m trace to nil to end of hole.	Casing sealed, good water return.	24 eighty-pound bags used.
EKAV-15-010	Baby Zone: Upgrading resources	2-3% mineralization from 78.2-158.5 m, in grey granite then nil to trace in sericitic granite to end of hole. Cassiterite evident in mineralized zone.	Lost water temporarily 23 m, returned, but not 100%. Total loss 72m-146 m, then partial return. Put in plug at 21 m and cemented to top.	4 eighty-pound bags used. Put in plug at 21 m and cemented to top.
EKAV-15-011	Baby Zone: Upgrading resources	Altered Meguma to 85 m, minor mineralization. 85-113.5 m granite with local Transition Zone, 2-4% po/cpy/py/sph. Rare moly/flu. 113.5-122 m trace min – py/sph/possible cas.	Hole 10, lost water return at 23 m with partial return below, intermittent.	4 eighty-pound bags used. Put in plug at 21 m and cemented to top.
EKAV-15-012	Baby Zone: Upgrading resources	Scarce mineralization to 62 m, then 1-2% sulphides to 99.5 m. 99.5-108 m mixture of Meguma/granite/Contact Zone with 1% sulphides. 108-112.5 m granite with 2% po/cpy. 112.5-131.5	Lost water 12 m, partial return, intermittent.	Plugged at 10 m and cemented. 2 eighty-pound bags used.

DDH	Objective	Comments Mineralization	Comments: Water, Etc.	Comments: Cementing
		m tr min. 131.5-134 m 2-3% po/cpy/sph/py. 134 m-to end of hole minor mineralized sections comprise 20% of total.		
EKAV-15-013	Baby Zone: Upgrading resources	Trace mineralization in Meguma to 38 m, then minor mineralization to 42 m, then 2-5% po/cpy/aspy/sph/py to 86 m. Cas at 45.8 m. Granite start 86 m, 2-3% po/py/cpy/sph to 109 m. 109-132.2 m 1% mineralization. 134.2-142.3 m granite 1-2% sulphides. 142.3-145.5 m Meguma. 145.5-153.5 m granite 1% sulphides. 153.3-155 m Meguma. 155-161 m granite, trace min, cas/wolf? in band 156 m.	Lost water 25 m, minor intermittent return.	Plugged and cemented at 23 m, 4 eighty-pound bags used.
EKAV-15-014	Baby Zone: Upgrading resources	Trace mineralization in Meguma to 60 m, then 0.5% sulphides to 86.5 m then 1-2%. From >94 m mineralization and alteration decreases. Trace mineralization at bottom of hole.	Lost water 5 m, no return throughout hole.	Plugged and cemented at 20 m, did not come to surface. 4 eighty-pound bags used.
EKAV-15-015	Baby Zone: Upgrading resources	Trace mineralization in Meguma to 71.3 m. 1.3-76.5 m trace mineralization in sericitic granite. 76.5-85 m non-sericitic with 2-3% py/cpy/po (cas 81.2 m). 85-161 m sericitc granite local cpy/py/po/sph - trace overall. Scattered py cubes throughout. 161-166 m Meguma tr py/moly. 166-175 m non-mineralized sericitic granite, 175-180.5 m slightly sericitic granite, 1% po/py/cpy/sph. 180.5-183 m non-min. 183-188 m partial Meguma 1-2% po/py/cpy/tr cas. 188-194.5 m Meguma/CZ 1-2% po/py/cpy. 194.5-199.5 m Meguma 1 - 2% po/py/cpy/cas. 199.5-201 m non-sericitic granite 2-5% po/cpy/sph. 201-206 CZ/Meguma/granite <0.5% min. 206-238.5 m Meguma, trace min. 238.5-251m light grey granite, 1-2% po/sph/flu 243-246.5 m, minor cas/py 246.5-247.5 m.	Good water return throughout hole, minor loss briefly several times.	Cemented from bottom up, 38 eighty-pound bags used.
EKAV-15-016	Baby Zone: Upgrading resources	<0.5% aspy/cpy/py/cas to 19.9 m, trace min 19.9-44 m, 1% py/cpy/sph/po/cas 44-55 m, unmineralized 55-65 m except minor local cpy/cas. 65-101 m mineralized in 1-3m intervals, approx 30% of total with up to 5% cpy/sph/py/cas/moly. 101-164 m trace cpy/sph/py/cas, higher in two 1 m zones.	Good water return throughout hole.	Cemented from bottom up, 25 eighty-pound bags used.
EKAV-15-	Baby Zone:	Trace to 0.5% py/cpy, local sph, cas,	Water loss 27 m,	Cemented from

DDH	Objective	Comments Mineralization	Comments: Water, Etc.	Comments: Cementing
017	Upgrading resources	flu, moly, local 1-4 m zones of darker granite with higher percent mineralization.	tried Gstop and then cemented, waited 24 hours to redrill and got water return. Water return diminished below 61 m but returned with additional mud. Good return thereafter to end of hole.	bottom up, 23 eighty-pound bags used.

Duck Pond Zone: Drill holes DPAV-15-020 to 024 were targeted on the Duck Pond Zone. This zone had not been drilled to a formal resource level previously, and the drilling in this case had the objective of elucidating the geological model of the deposit. As a result, all holes were relatively close spaced in order to increase ability to correlate from hole to hole (see Table 10.10).

Table 10.10
Summary of Drill Holes at the Duck Pond Zone

DDH	Objective	Comments Mineralization	Comments: Water, Etc.	Comments: Cementing
DPAV-15-020	Geological model and increasing resources	Minor py/sph in greywacke units, and qtz veins. Minor cas in both gwke and arg. Trace cpy/ga. Overall trace to 0.5% mineralization.	Lost water return 30 m, intermittent return after pushing casing to 18.5 m, approximately 150 m.	No cement, Vanruth plug inserted 30 m, casing left in (22.5 m + 2-foot top extension).
DPAV-15-021	Geological model and increasing resources	Trace pyrite localized in bands and stringers, possible rare yellow sph to 198.3 m. 198.3-204.8 m 1% py/cas concentrated in 2-10 cm bands in gwke. 1% py/cas as previous 221-237.3 m. Trace py/cas/sph/po 237.3-275 m	Lost water return 26 m permanently.	40 eighty-pound bags used, unsure of cement level.
DPAV-15-022	Geological model and increasing resources	Trace py locally, py/cas bands noted 35, 73, 111.5, 115, 170.2 m	Water return entire hole.	35 eighty-pound bags used, unsure of cement level.
DPAV-15-023	Geological model and increasing resources	Cas often with py/sph and sometimes cpy 86.5-120.5 m and 138-157 m, minute amounts elsewhere.	Water return entire hole.	26 eighty-pound bags used, unsure of cement level.
DPAV-15-024	Geological model and increasing resources	Trace local py/cas to 125 m. 125-180 m numerous cas/sph/py bands and stringers, trace cpy locally. Trace local py/sph/cas 180-227 m.	Water return entire hole.	24 eighty-pound bags used, hole plugged at 30 m.

Main Zone: Finally, seven drill holes were completed on the Main Zone, two being from the northwest at the southwest end of the zone, mainly with the objective of verifying historic drilling, and five in the northeast of the zone with the objective of upgrading and increasing resources (see Table 10.11).

Table 10.11
Summary of 7 Holes Drilled on the Main Zone

DDH	Objective	Comments Mineralization	Comments: Water, Etc.	Comments: Cementing
EKAV-15-018	Main Zone: upgrading and increasing resources	Trace aspy/py/po in Meguma. In granite section 1% cpy/sph/cas 103.5-111 m. 111-123.5 m <0.5% po/sph/py/cas, localized in dark grey sections. 123.5-145.5 m localized py/cpy/cas, trace overall. 145.5-160 m localized sph/py/flu/cas/moly, trace overall. 160-182 m localized cpy/py/cas, trace overall.	Lack of water return. Tried extending casing and adding Gstop, but only temporary partial return. Return established 45 m and lost completely at 56 m. No return for rest of hole. Numerous fracture zones and shears throughout.	25 eighty-pound bags used, unsure of cement level.
EKAV-15-019	Main Zone: upgrading and increasing resources	Trace cpy/aspy/py/aspy/sph in Meguma, local concentrations in granite of po/py/aspy/cas/sph/cpy. Overall trace to 0.5%	Variable water return initially, casing extended to 13.5 m and good return after. Partial loss 87-90 m, no return after 145 m to end of hole.	40 eighty-pound bags used, unsure of cement level.
EKAV-15-025	Main Zone: upgrading and increasing resources	Trace py/cpy/cas/flu to 84.5 m. Most mineralization in minor dark grey units. 84.5-108.7 m 0.5% py/aspy/cas. 108.7-117 m 3- 5% py/ga/aspy/cpy/cas. 117-129 m 1% py/cas/sph. 129-133.5 m tr py. 133.5-143 m 3-5% py/sph/cas. 143-155 m 0.5% py/cas/sph. 155-173.5 m trace to <0.5% py/cpy/sph/cas. 177-223.5 m 0.5% cpy/py/cas/sph/aspy, locally 2%. 223.5-245 m py/po/sph in dark grey granite zones only (10% of volume), trace overall. Cas noted at 244 m.	Water return entire hole.	39 eighty-pound bags used, unsure of cement level.
EKAV-15-026	Main Zone: upgrading and increasing resources	Mineralization restricted to dark grey zones and varies from 2-5% cpy/py/sph with local po/cas. Percentage of mineralized zones varies from 5-40% and mineralization averages on a larger scale from trace to 2%. Best continuous section is 192-198.2 m with 5% py/cpy/cas. Local trace py/sph/po 219.3-230 m.	Water return entire hole.	37 eighty-pound bags used, unsure of cement level.

DDH	Objective	Comments Mineralization	Comments: Water, Etc.	Comments: Cementing
EKAV-15-027	Main Zone: upgrading and increasing resources	As above, but best continuous section 170-173.5 m with similar mineralization. Likely the same unit.	Water return entire hole.	34 eighty-pound bags used, unsure of cement level.
EKAV-15-028	Main Zone: upgrading and increasing resources	Similar to hole 26. Best continuous mineralized section 159.6-162.5 m, lower grade equivalent 190-200 m.	Lost water 12.5 m, greased rods and got return back, but only partial return shortly after. Little return to bottom of hole.	36 eighty-pound bags used, unsure of cement level.
EKAV-15-029	Main Zone: upgrading and increasing resources	Few continuous mineralized zones. 101-104.1 m, 108.8-116.3 m (40% zones of min.), 130.5-132 m, 165.3-168. 5 m. Mostly 5-10% zones of mineralization.	Return throughout hole, partial leakage out of hole 28 casing.	42 eighty-pound bags used, unsure of cement level.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE PREPARATION

11.1.1 2014 Drill Program

Core was placed in numbered and marked core boxes at the drill site and a quick log prepared. After the quick logging was completed the core boxes were covered and taped closed with duct tape, removed from the drill site each evening by Avalon personnel and transferred to the core logging area. At the core logging area, boxes were opened, drill core was logged in detail and marked for sampling and all core photographed prior to sampling.

Sampling was typically undertaken on 1.5 m intervals within mineralized sections. Only a limited amount of sampling was undertaken in non-mineralized sections (limestone and greywacke). Core was split using a manual core splitter, with the remaining ½-core reassembled in order in the core box. Sample material was placed in plastic sample bags with sample number marked on the outside of the bag and a sample tag stapled to the inside fold of the bag. A duplicate tag was placed in the core box to mark the sample interval and the sample interval was also noted on the core box divider. Duplicate samples were obtained from drill core by splitting core in half, with one half noted as the main sample and the other half noted as the duplicate in the sample log. Standards and blanks were inserted in the sample list on a pre-determined basis.

Bagged samples were placed in 20-L plastic pails. The pails were sealed with secure lids and taped closed and the sample numbers noted on the outside of the pail.

Once a sufficient number of samples had been prepared, samples were shipped by courier to ALS Canada Ltd. (ALS) in Sudbury for initial sample preparation. After initial sample preparation, ALS shipped the samples to its Vancouver facility for assaying. Sample preparation at ALS involved the following:

1. Weighing of samples as received.
2. Sample log-in with bar code.
3. Coarse crushing to 70% <3.36 mm.
4. Sample split using rotary splitter.
5. Duplicate split using rotary splitter (for duplicates at ALS).
6. Triplicate split using rotary splitter (for check samples to other laboratories).
7. Pulverize 100 gm split to 95% passing 75 µm.
8. Crushing and pulverizing quality control (QC) tests (internal ALS QC check).

Coarse check sample splits were shipped by ALS to SGS Canada in Lakefield, Ontario, and to Activation Laboratories Ltd. (Actlabs) in Ancaster, Ontario, under the same sample number as the main sample with an identifier letter code. Sample preparation of the coarse check samples included sample log-in and pulverization to 95% passing 75 µm. Splits were prepared using a rotary splitter. Two standards were inserted in the check sample batches.

A total of 404 samples (excluding the 57 duplicates, standards and blank samples) were submitted to ALS for multi-element work using the ME-MS81, MEOG62, ME-ICP06 and Fusion XRF for Sn methods. Sixteen blanks, 15 standards and 13 field duplicates were inserted into the three sample shipments to monitor contamination, accuracy and precision. In-house Avalon standards STD-292 and SDT-300 were cuts prepared for Avalon by ALS from blended and homogenized utilized East Kemptville mineralized core material from original test work to try and cover a range of potential indium and tin grades. Certified reference material was obtained from both CANMET Mining and Mineral Sciences Laboratories (CANMET) and Ore Research and Exploration Pty Ltd (ORE). The details of these standards, including certification, are accessible on the respective websites.

The CANMET standard utilized was MP-1B, "certified reference material for zinc-tin-copper lead ore" and contains 1.61% Sn. The reference material is ore from Mount Pleasant, New Brunswick.

The standards supplied by ORE were OREAS-140, 141, and 142 tin standards. These standards are derived from a tin oxide deposit in New South Wales, Australia, but despite the differing mineralogy to East Kemptville, were utilized as the only readily commercially available tin rock standards at suitable grades. They, along with MP-18, were used to in turn to certify Avalon's new standards through a Round Robin process.

Blanks were derived from barren greywacke drill core taken from hole GM-10-08, previously drilled by Avalon in the Gardener's Meadow exploration area of the East Kemptville claims.

Sample preparation work on the samples at ALS involved coarse crushing of drill core samples to a minimum of 70% <3.36 mm (6 mesh) then pulverizing a 1,000 g split to better than 95% passing minus 75 µm. Table 11.1 summarizes the sample program for the 2014 drill program.

SGS Canada, ALS and Actlabs are independent of Avalon.

Table 11.1
Summary of Samples for 2014 Drill Program

Sample Type	Number	% of Main Samples
Main Samples	404	100
Duplicates	26	6.44
Standards	15	3.71
Blanks	16	3.96
Total	**461**	

11.1.2 2015 Drill Program

For the 2015 drill program, the sample treatment at the core logging facility was similar to 2014, with the exception that the samples were shipped to Actlabs' sample preparation facility in either New Brunswick or Ontario, with the New Brunswick facility utilized except in some cases where a backlog had built up in New Brunswick. In the latter case, the samples were shipped direct to the Actlabs laboratory in Ancaster, Ontario.

At the drill site, core was placed in numbered and marked core boxes and a quick geological log prepared. Also completed at the drill site was a simple geotechnical log including recovery and RQD. A limited number of holes were also logged geotechnically in detail, some before splitting and some after splitting, depending on the availability of the geotechnical logging geologist.

After geotechnical logging at the drill site, core boxes were covered and taped closed with duct tape, removed from the drill site each evening by Avalon personnel and transferred to a secure storage area. At the core storage area, boxes were opened, drill core was logged in detail and marked for sampling and all core photographed prior to sampling.

Over three thousand (3,000) assay samples were obtained by sawing half core utilizing an electric core saw. Sampling was typically undertaken on 1.5 m intervals within mineralized sections though, depending on geological boundaries, there may be some variation in sample length.

For the 2014 program, only a limited amount of sampling was undertaken in non-mineralized sections (limestone and greywacke). However, in 2015, in light of the occurrence of mineralization to the boundary of sampling, prior to the start of the 2015 drill program, additional sections of unsampled 2014 core were split and assayed and, in some cases, contained significant mineralization that was contiguous with existing known mineralization.

As mentioned above, core was split using an electric core saw, with the remaining ½ core re-assembled in order in the core box. Sample material was placed in plastic sample bags with sample number marked on the outside of the bag and a sample tag stapled to the inside fold of the bag. A duplicate tag was placed in the core box to mark the sample interval and the sample interval also noted on the core box divider.

Duplicate samples were split from drill core by ½ splitting core, with one half noted as the main sample and the other half noted as the duplicate in the sample log. Standards and blanks were inserted in the sample list on a pre-determined basis. The statistics for samples, duplicates, blanks and standards for Actlabs are given in Table 11.2 and Table 11.3 below. As can be seen, Avalon inserted a total number of control samples (standards and blanks) of one to every nine samples.

Table 11.2
Total Sample Count

Laboratories	ACT_ANC
No. of Batches	20
No. of DH Samples	2438
No. of Field duplicates of drill core	76
No. of Standard Samples	347

Table 11.3
Standard Insertion Statistics

Standard Type	DH Sample Count	Standard Type Count	Standard Sample Count	Ratio of QC Standard to DH Samples
Avalon	2438	4	262	1:9
Laboratory	2438	6	85	1:29

Bagged samples were placed in 20 litre plastic pails. The pails were sealed with secure lids and taped closed and the sample number intervals noted on the outside of the pail. Once a sufficient number of samples had been prepared, samples were shipped by courier to Actlabs in Ancaster, Ontario, or the Actlabs sample preparation facility in Fredericton, New Brunswick.

The core logging geologist was responsible for making a determination as to whether a sample was potentially mineralized or likely to be waste rock in any open pit. The instructions sent to the laboratory were different according to whether the sample was designated in the first, mineralized, or second, unmineralized, category.

Coarse check sample splits were prepared and then shipped by Actlabs (the “primary” lab) to Avalon’s office in Toronto. Avalon inserted standards and then shipped the check samples to ALS laboratory (the ‘secondary” lab) in Vancouver, B.C., for analysis under the same sample number as the main sample with an identifier letter code. Sample preparation of the coarse check samples included sample log-in and pulverization to 95% passing 75 microns. Splits were prepared using a rotary splitter.

11.1.3 Low Grade Stockpile Surface Sampling Program

In order to verify the metal grade of the low-grade stockpile, a surface sampling program was completed in 2015. It is recognized that surface sampling has significant potential for error as the samples might not be representative of the internal material of the pile. However, the tin grade of the pile estimated by RAL from more than a thousand blasthole points was within 17% of the average grade from five RAL surface samples. Therefore, it was considered that a surface sample program would be sufficient for initial verification of grades for the purpose of classifying the resource as an Inferred Mineral Resource.

A program was completed with two samplers to reduce sample bias, each independently taking a sample at points at 50 m intervals across the length and width of the low-grade stockpile, plus samples around one side of the bottom of the pile. The two samples from the two individuals from each site were kept separate for analysis in order to investigate any sampling bias on the part of one or other sampler. A total of approximately 270 kg was collected with each sample being about 5 kg.

These results are compared with the RAL Closure Plan (RAL, 1993) data in Table 11.4 and show that the Avalon estimates for Sn (0.117% Sn) and Zn are within 11% of the surface samples quoted by RAL (0.106% Sn) and, for Sn, higher than the blasthole assays estimate or 0.09% Sn given by RAL (1993). Copper is close to the block model estimate and below the RAL surface sample estimates. These analytical results are considered to be in reasonable agreement with the average resource grade quoted by RAL (1993) and support the estimated grades of the low-grade stockpile.

Table 11.4
Comparison of Avalon (2015) and RAL (1993) Estimated Grades for the Low-Grade Stockpile

Dump	Tonnes (Mt)	Surface Area (ha)	Sample Description	Grade (%)		
				Sn	Zn	Cu
Low Grade Stockpile (RAL, 1993)	5.87	22.7	Ave LG1-5	0.108	0.106	0.064
Sampled on Surface (RAL)[1]			Mean	0.106	0.106	0.068
Block Model Estimate (RAL)[2]			Mean	0.091	0.153	0.058
Mean 2015 Sample Assays (Avalon)	5.87			0.117	0.094	0.054
Differences						
Sampled on Surface (RAL)				100%	100%	100%
Block Model Estimate (RAL)				86%	144%	85%
This study (Avalon)				110%	89%	79%

[1] Measured by RAL in samples collected from surface piles in 1991.
[2] Estimated by RAL from more than 1,000 sample analyses from blasthole cuttings.

In 2016, Avalon completed an additional sampling program of the low-grade stockpile in order to obtain a sample for metallurgical testwork. The total material collected was 122 samples totalling 320 kg. The analyses of the samples were completed by Bureau Veritas utilizing method LF100 (Lithium metaborate fusion followed by ICP-MS).

The results of this 2016 sampling give an average grade of 0.1273% Sn in the 320 kg of sample. These results are considered by Avalon to be in close agreement with previous sampling by Avalon and RAL with the overall average slightly higher than previous. Thus, it is considered that this sampling is additional confirmation of the grade of the surface of the Low-Grade Stockpile.

11.2 SAMPLE ANALYSIS

Samples in 2014 were shipped to ALS (Vancouver, BC) for analysis and check samples returned to Avalon. Avalon then inserted standards in the sequence of check samples and shipped them to SGS (Peterborough, Ontario) or Actlabs (Ancaster, Ontario) for check analysis. In 2015, the initial processing and analysis of the drill core samples was completed by Actlabs (Ancaster, Ontario) and the check samples sent to ALS (Vancouver, BC) for analysis.

The low-grade stockpile samples collected from each site were shipped to Actlabs for rush analysis including multi-element Ultratrace-7 (56 elements) and XRF for Sn (plus 19 elements including whole rock analysis).

11.2.1 ALS

Sample analysis at ALS included the following procedures:

1. Loss on ignition determination at 1,000°C (method OA-GRAO5).
2. Multi-element assay by lithium borate fusion/ICP-MS (method MEMS81) (31 elements).
3. Base metal assay by four-acid digestion and ICP-AES (Method ME-4ACD81 as part of ME-MS81).
4. Major oxides by ICP-AES (method ME-ICP06).
5. Four-acid digestion/ICP-AES (method ME-OG62) for Cu, Pb and Zn.
6. Sn by XRF fusion, ore grade (method Sn-XRF10).
7. XRF fusion, other elements, ore grade by method ME-XRF10.
8. Loss on Ignition for ME-XRF06 (method OA-GRA06).

Method ME-MS81 involves fusion of the sample in a lithium borate/lithium tetraborate flux, dissolution of the melt in a three-acid mixture (nitric, hydrochloric and hydrofluoric) and analysis of the solution by ICP-MS. Method ME-4ACD81 uses a four-acid digestion to place the elements into solution. Four-acid digestion followed by ICP-AES is used for high grade base metal analysis. Fusion XRF is used for samples over limits using method ME-MS81.

The limits of detection for method ME-MS81 and ME-4ACD81 are noted in Table 11.5 and Table 11.6.

Table 11.5
Limits of Detection for Method ME-MS81

Element	Symbol	Unit	Lower Limit	Upper Limit
Barium	Ba	ppm	0.5	10,000
Cerium	Ce	ppm	0.5	10,000
Chromium	Cr	ppm	10	10,000
Cesium	Cs	ppm	0.01	10,000
Dysprosium	Dy	ppm	0.05	1,000
Erbium	Er	ppm	0.03	1,000
Europium	Eu	ppm	0.03	1,000
Gallium	Ga	ppm	0.1	1,000
Gadolinium	Gd	ppm	0.05	1,000
Hafnium	Hf	ppm	0.2	10,000
Holmium	Ho	ppm	0.01	1,000

Element	Symbol	Unit	Lower Limit	Upper Limit
Lanthanum	La	ppm	0.5	10,000
Lutetium	Lu	ppm	0.01	1,000
Niobium	Nb	ppm	0.2	2,500
Neodymium	Nd	ppm	0.1	10,000
Praseodymium	Pr	ppm	0.03	1,000
Rubidium	Rb	ppm	0.2	10,000
Samarium	Sm	ppm	0.03	1,000
Tin	Sn	ppm	1	10,000
Strontium	Sr	ppm	0.1	10,000
Tantalum	Ta	ppm	0.1	2,500
Terbium	Tb	ppm	0.01	1,000
Thorium	Th	ppm	0.05	1,000
Thallium	Ti	ppm	0.5	1,000
Thullium	Tm	ppm	0.01	1,000
Uranium	U	ppm	0.05	1,000
Vanadium	V	ppm	5	10,000
Tungsten	W	ppm	1	10,000
Yttrium	Y	ppm	0.5	10,000
Ytterbium	Yb	ppm	0.03	1,000
Zirconium	Zr	ppm	2	10,000

Table 11.6
Limits of Detection for Method ME-4ACD81

Element	Symbol	Unit	Lower Limit	Upper Limit
Silver	Ag	ppm	0.5	100
Arsenic	As	ppm	5	10,000
Cadmium	Cd	ppm	0.5	1,000
Cobalt	Co	ppm	1	10,000
Copper	Cu	ppm	1	10,000
Lithium	Li	ppm	10	10,000
Molybdenum	Mo	ppm	1	10,000
Nickel	Ni	ppm	1	10,000
Lead	Pb	ppm	2	10,000
Scandium	Sc	ppm	1	10,000
Zinc	Zn	ppm	2	10,000

11.2.2 SGS Canada Assays

Sample analysis at SGS Canada was by method GE-ICM90A. This method is a sodium peroxide fusion followed by acid dissolution and assay using combined ICP-AES and ICP-MS for 55 elements. The limits of detection for the method are as noted in Table 11.7.

Table 11.7
Limits of Detection SGS Method GE-ICM90A

Element	Detection Limits	Element	Detection Limits	Element	Detection Limits
Ag	1-1,000	Ge	1-1,000 ppm	Sb	0.1-10,000 ppm
Al	0.01-25%	Hf	1-10,000 ppm	Sm	0.1-1,000 ppm
As	0.0005-10%	Ho	0.05-1,000 ppm	Sn	1-10,000 ppm
Ba	0.5-10,000ppm	In	0.2-1,000 ppm	Sr	0.1-10,000 ppm
Be	5-2,500 ppm	K	0.1-25%	Ta	0.5-0,000 ppm
Bi	0.1-1,000 ppm	La	0.1-10,000 ppm	Tb	0.05-1,000 ppm
Ca	0.1-35%	Li	0.001-5%	Th	0.1-1,000 ppm
Cd	0.2-10,000 ppm	Lu	0.05-1,000 ppm	Ti	0.01-25%
Ce	0.1-10,000 ppm	Mg	0.01-30%	Tl	0.5-1,000 ppm
Co	0.5-10,000 ppm	Mn	0.001-10%	Tm	0.05-1,000 ppm
Cr	0.001-10%	Mo	2-10,000 ppm	U	0.05-1,000 ppm
Cs	0.1-10,000 ppm	Nb	1-10,000 ppm	V	5-10,000 ppm
Cu	5-10,000 ppm	Nd	0.1-10,000 ppm	W	1-10,000 ppm
Dy	0.05-1,000 ppm	Ni	5-10,000 ppm	Y	0.5-1,000 ppm
Er	0.05-1,000 ppm	P	0.01-25%	Yb	0.1-1,000 ppm
Eu	0.05-1,000 ppm	Pb	5-10,000 ppm	Zn	5-10,000 ppm
Fe	0.01-30%	Pr	0.05-1,000 ppm	Zr	0.5-10,000 ppm
Ga	1-1,000 ppm	Rb	0.2-10,000 ppm		
Gd	0.05-1,000 ppm	Sc	0.0005-5%		

11.2.3 Actlabs Assays

Assaying of coarse duplicate samples in 2014 or initial analysis in 2015 at Actlabs was by method FUS-MS-Na_2O_2, a sodium peroxide fusion, followed by acid dissolution and ICP-MS finish. Tin and tungsten, if high grade and over limits for ICP, were rerun using the XRF method.

In detail, samples classified as mineralized were routinely analysed by Actlabs methods as follows:

1. Weighing of samples as received.
2. Sample log-in with bar code.
3. Coarse crush ½ core (about 4 kgs) to 90%, -6 mesh, or <3.36 mm.
4. Sample split out 1 kg using rotary splitter.
5. Crush the 1 kg sample to -10 mesh, or ~2mm.
6. Riffle-split and pulverize 250 g to 95% passing 200 mesh.
7. Pulverize a 125 g duplicate split every ten samples to prepare a duplicate to be shipped to the secondary laboratory.
8. Analysis of the pulp by methods Ultratrace 7 plus 1E-Ag, Whole Rock Package 4C plus XRF Sn and W. Any Cu or Zn over limits, run 8-peroxide analyses. Also, complete 4C-C, S.

The purpose of the initial coarse crush at -6 mesh was in order to preserve sample suitable for a future metallurgical pilot plant. Normal -10 mesh preparation for assaying purposes would provide sample too fine grained for metallurgical testing.

For samples considered unmineralized, the approach was slightly different:

1. Weighing of samples as received.
2. Sample log-in with bar code.
3. Coarse crush ½ core (about 4 kgs) to 90%, -10 mesh, or 2 mm.
4. Riffle split and pulverize weighed 100 g to 95% passing 200 mesh.
5. Combine three pulverized contiguous drill core samples to make one 300 g sample (in cases where three contiguous samples were not present, then either one or two samples were utilized).
6. These samples were then prefixed with "COMP" and the number of the last sample in the sequence.
7. Analysis of the pulp by methods Ultratrace 7, Whole Rock Package 7 and 4F-C, S.

For unmineralized samples, combining typically three samples at a time reduced analytical costs, but still provided continuous geochemistry aimed at modelling acid generation potential of any waste rock to be mined from future open pits.

Method Ultratrace 7: This method utilizes sodium peroxide fusion, followed by ICP-MS or ICP. This method provides analyses for 56 elements, including Sn, In, Zn, Cu and Li. However, its maximum level for base metals is 10,000 ppm or 1%.

Method Whole Rock 4C: This method utilizes lithium metaborate/tetraborate fusion followed by XRF to provide the standard suite of major rock forming elements plus Loss on Ignition (LOI). Analysis of Sn and W by XRF was added to this package. Note that early in the program Actlabs reported tin as SnO_2 in %, not Sn in ppm. At Avalon's request, this was changed partway through the program.

Method Code 8-Peroxide: This analysis utilizes sodium peroxide fusion and ICP-MS similar to Ultratrace 7 but adjusted to provide high upper limits for Cu and Zn.

Method 4F-C, S: An add on to method 4C using solid state infrared absorption.

11.3 LABORATORY CERTIFICATIONS

ALS laboratories in Sudbury and Vancouver are ISO 9001 certified. The Vancouver laboratory holds ISO 17025 certification for most procedures, including EMS81 and XRF10. The laboratories follow a well-documented internal quality assurance/quality control (QA/QC) protocol for control of sample preparation and for assay quality control and quality assurance. Internal assay QA/QC procedures include the insertion of certified blanks and standards in each assay batch and the analysis of internal laboratory duplicates and replicates at a rate of 5% of the assay samples.

SGS Canada at Lakefield holds ISO 17025 certification for laboratory quality assurance and quality control. The laboratory follows a standardized protocol for insertion of certified standards and blanks, internal duplicates and replicates.

Activation Laboratories Ltd. (Actlabs) is ISO 17025 accredited and/or certified to 9001: 2008. It is also a Standards Council of Canada accredited laboratory (Accreditation Number 0266) and follows a standardized protocol for insertion of certified standards and blanks.

Natural Resources Canada (NRCan) helps improve the reliability of measurements at mineral analysis labs in Canada and around the world by certifying reference materials. Certified reference materials—Canadian ores, rocks, concentrates, tailings, soils, alloys, anodes and radioactive materials—are used as standards to validate analytical measurement methods or to calibrate analytical instruments used in analytical chemistry. These standards support decision-making around mineral resource production, commodity valuation and compliance with environmental regulations. NRCan's analysis and certification is conducted through its Canadian Certified Reference Materials Project (CCRMP) run by CANMET, the NRCAN laboratory.

Ore Research & Exploration Pty Ltd (ORE) is an Australian-based company specialising in the development of low cost certified reference materials (CRMs) for the mining and exploration industry since 1988. ORE combines a blend of innovative technology with world's best practices to maintain its position as a leading producer of reference materials.

In the opinion of the author, the standard QA/QC procedures employed by each of the sample preparation and assay laboratories are acceptable.

11.4 SAMPLE SECURITY

The sample security measures employed included a documented chain of custody and shipment of samples in secure packaging by reputable courier companies. Sample receipt notifications were provided by each laboratory to Avalon and no discrepancies between the sample lists and the receipt notices were noted. Sample security at the drill rig and at the core logging facility was deemed to be adequate for the type of samples and no deficiencies in the drill core handling and sample storage procedures were noted.

Overall, the author is satisfied that the sample preparation, sample assaying and sample security procedures employed were adequate for the purposes and no deficiencies were noted.

12.0 DATA VERIFICATION

The following section outlines the procedures adopted for the processing of drill core from the 2014 and 2015 drill programs used as the basis for estimating the mineral resources.

Methodology and practices applied for sampling of the low-grade ore stockpile are summarized in Section 11.1.3.

12.1 DATA VERIFICATION – HISTORICAL DATA AND 2014 DRILL PROGRAM

Data verification for the 2014 drill program and the historical data in the resource database included the following:

1. Comparison of 2014 drill core sample numbers against assay sample shipment lists and sample receipt list. One discrepancy in the original sample numbering sequence for the 10 mesh check samples was noted. This appears to be a sequence numbering error in the shipment list which was corrected by the receiving laboratories.
2. Drill collar coordinates were surveyed in by a qualified Nova Scotia land surveyor (Scotia Surveys) using a Magellan Ashtech ProMark2 static GPS equipment. The base station occupied Nova Scotia High Precision Network monument #229333. Survey data was recorded in Nova Scotia 3° M.T.M. projection, Zone 5, NAD 83, ATS 77 Coordinates and converted to NAD 27 UTM coordinates by Scotia Surveys to match the coordinate system used in the database.
3. Site visit and inspection of 2014 drilling procedures, core logging, and sampling for the 2014 drill program by Hains Engineering.
4. Collection of due diligence ¼-core samples and independent assaying of samples from holes EKAV 14-001 through EKAV 14-004 of the 2014 drill program by Hains Engineering.
5. Comparison of 2014 drill core assays against assay certificates. PDF assay certificates were manually checked against Excel copies of assay certificates and against the sample data base.
6. Comparison of historic drill logs and assay certificates against the historic Excel database used in the resource estimate.
7. Verification of historic Rio Algom Limited (RAL) QA/QC data detailed in the 1983 feasibility study against the historic drill logs and the resource database.
8. Inspection of selected historic drill core stored at the NSDNR Core Library in Stellarton, Nova Scotia and verification of descriptions in historic drill logs by Hains Engineering.
9. Collection of due diligence ¼-core samples and independent assaying of samples from selected drill core intervals of historic drill core stored at Stellarton by Hains Engineering.

The due diligence' check samples collected from the Stellarton historic drill core were assayed at SGS Lakefield. The assay results are shown in Table 12.1.

Table 12.1
Due Diligence Check Samples, Stellarton Historic Drill Core Storage Library

Hole No.	Interval		Sample Number	Assay		
	From (m)	To (m)		Sn (%)	Cu (%)	Zn (%)
89-229	7.0	10.0	20734	1.024	0.19	1.94
	10.0	13.0	20735	1.118	0.30	0.68
	7.0	**13.0**		**1.07**	**0.25**	**1.31**
Due Diligence Samples	**¼ core**					
	7.0	8.5	B00010551	0.63	0.16	1.77
	8.5	10.0	B00010552	0.29	0.17	1.02
Length Weighted	7.0	10.0		0.46	0.17	1.40
	10.0	11.5	B00010553	1.64	0.08	0.34
	11.5	13.0	B00010554	0.49	0.34	0.95
Length Weighted	10.0	13.0		1.07	0.21	0.65
	7.0	**13.0**		**0.77**	**0.19**	**1.03**

Given the high heterogeneity of short sample intervals, especially relatively high-grade samples, the comparison of assay results is reasonable.

Due diligence samples collected by Hains Engineering from holes EKAV 14-001 through EKAV 14-004 were ¼-core samples. These samples were assayed at SGS Lakefield using peroxide fusion with an ICP-OES finish for Sn, Cu and Zn and an atomic absorption (AA) finish for In. Detection limits for this method are not directly comparable to other assay methods such as the methods used to assay the main samples at ALS Canada Ltd. (ALS) in Vancouver. The results of the due diligence assays and comparison with the original assays from ALS are detailed in Table 12.2.

The results are in broad agreement, although it is noted that the sampling differences, short sample intervals and the nuggety nature of the tin mineralization give rise to potential for wide variations in assay results, especially for tin.

The results of the data verification indicated no issues with drill core sampling, sample preparation and security and the 2014 drill program assay data base. Review by Hains Engineering of the historic drill core stored in Stellarton indicated that the historic drill logs adequately represented the historic drill core and that there is no reason to believe the historic drill logs misrepresent the reported mineralization. The data verification process has confirmed the historic assay database.

Due diligence sampling has confirmed the general tenor of mineralization in the samples analysed and the results give no reason to doubt the integrity of the main samples.

Table 12.2
Due Diligence Sample Assay Results

					Original Sample Assays				Due Diligence Check Sample Assays (Converted from g/t or %)			
Hole No.	**Sample No.**		**Sample Interval**		**Sn (ppm)**	**Cu (ppm)**	**Zn (ppm)**	**In (ppm)**	**Sn (ppm)**	**Cu (ppm)**	**Zn (ppm)**	**In (ppm)**
	Original	**Due Diligence**	**From (m)**	**To (m)**								
EKAV-14-001	R334002	B00010559	51.0	52.5	2,950	3,060	587	7.49	560	2,290	704	<20
	R334010	B00010557	85.0	86.5	2,510	3,180	281	16.15	1,200	5,500	387	20
	R334011	B00010558	86.5	88.0	1,275	778	1,420	9.28	1,500	921	1,050	<20
	R334040	B00010555	125.5	127.0	1,160	2,490	188	4.57	2,200	2,300	<300	<20
	R334041	B00010556	127.0	128.5	304	83	649	2.96	140	<300	539	<20
EKAV-14-002	R334085	B00010572	105.0	106.5	7,150	2,380	3,200	24	5,300	1,870	2,950	40
	R334086	B00010573	106.5	108.0	1,025	1,530	2,710	20.2	1,100	1,400	2,160	30
	R334099	B00010564	123.0	124.5	1,740	617	9,460	42.6	1,700	617	6,720	30
	R334100	B00010565	124.5	126.0	1,070	1,300	2,540	17.1	1,800	1,270	2,040	<20
EKAV-14-003	R334131	B00010566	51.0	52.5	1,445	339	125	2.04	1,100	<300	<300	<20
	R334132	B00010567	52.5	54.0	17,200	475	1,280	9.75	34,400	480	1,230	20
	R334176	B00010562	123.0	124.5	7,390	11	53	<0.05	6,700	<300	<300	<20
	R334177	B00010563	124.5	126.0	1,265	263	2,140	9.36	1,400	511	2,170	<20
	R334181	B00010560	130.5	132.0	390	93	1,880	6.48	960	<300	1,350	<20
	R334183	B00010561	132.0	133.5	1,730	183	1,270	5.11	2,500	<300	1,330	<20
EKAV-14-004	R334270	B00010568	159.3	160.55	472	346	>10,000	32.4	300	<300	11,000	50
	R334271	B00010569	160.55	161.2	214	459	>10,000	36.6	420	436	13,500	20
	R334258	B00010571	143.5	145.0	862	272	2,860	15.15	690	340	3,010	20

Note: Original assays at ALS, Vancouver, lithium borate fusion with ICP-MS finish, minus10 mesh split from ½-core. Due diligence assays at SGS Lakefield, sodium peroxide fusion with ICP-AES finish, ¼-core split.

In the opinion of author, the 2014 sampling and assay data and the historic drill hole and assay data, as represented in the resource database, are reliable and can be used in resource estimation.

12.2 QUALITY ASSURANCE/QUALITY CONTROL PROTOCOLS – 2014 DRILL PROGRAM

QA/QC measures employed for the 2014 drill program included the following:

1. Insertion of standards and blanks in the main sample batches at a rate of approximately 4%. Both certified reference materials and in-house standards were used. Blanks were in-house material.
2. Assays of coarse duplicates to check sample preparation procedures and laboratory precision.
3. Assays of pulp duplicates to laboratory analytical precision.
4. Coarse check samples assayed at two separate laboratories to check sample preparation procedures and analytical bias.
5. Insertion of certified standard reference materials in check sample assay batches.
6. Internal laboratory QA/QC protocols incorporating the use of certified standards and blanks and duplicate and repeat assays.

Table 12.3 summarizes the QA/QC protocols for the 2014 drill program.

Table 12.3
QA/QC Protocols – 2014 Drill Program

Sample Type	Number	% Samples
Main Samples	404	100%
Coarse Duplicates	13	3.2% of main
Standards	15	3.7% of main
Blanks	16	3.9% of main
Check Samples – SGS	46	11.4% of main
Standards	4	8.7% of checks
Check Samples – Actlabs	46	11.4% of main
Standards	4	8.7% of checks

The standards used included a Certified Reference Material produced by CANMET (MP-1b-2208, a Sn-Cu-Zn-Pb standard) and high and low-grade tin certified reference standards (OREAS 140 and 142) produced by OREAS Pty. of Australia. In addition, two in-house standards (EKD-292 and EKD-300) and an in-house blank originally prepared as part of Avalon's 2007-2008 re-sample program, were supplied by Avalon. All standards were used in ALS assay samples, as were the Avalon-produced blanks. The OREAS standards were used with the check samples assayed by SGS Lakefield and Actlabs.

Information on the preparation of Avalon standards 292 and 300 is provided in Section 11.1.1. Data showing the chemical characterization of the Avalon in-house standards are detailed in Table 12.4. The grade range of the standards (2.41% Sn for 292 and 0.73% Sn for 300) gave a good representation of the high-grade ranges of mineralization at East Kemptville, with low relative standards deviations. The blank material with average tin value of 28.3 ppm and a standard deviation of 21.6 ppm is well suited to detecting contamination in the analysis process on typical mineralized samples with Sn grades around 1,500 ppm.

Table 12.4
Statistical Analysis – Avalon Produced Blanks and Standards – 2007/8 Re-Sample Program
(Values in ppm)

Blanks								
	Sn	**Zn**	**Cu**	**In**	**Ga**	**Ge**	**W**	**Ag**
Mean	28.31	192.13	39.44	0.24	12.9	1.8	1.9	0.7
SD	21.62	292.67	25.37	0.17	1.63	0.43	1.29	0.50
Variance	467.434	85,654.859	643.840	0.030	2.652	0.188	1.652	0.246
Number	8	8	8	8	8	8	8	8
Standard 292								
	Sn	**Zn**	**Cu**	**In**	**Ga**	**Ge**	**W**	**Ag**
Mean	24100	5290	411.67	1.62	25.2	14	841.2	1
SD	1,100.00	101.98	1.25	0.47	1.03	0.00	16.02	0.00
Variance	1,210,000.000	10,400.000	1.556	0.224	1.056	0.000	256.722	0.000
Number	2	3	3	3	3	3	3	3
Standard 300								
	Sn	**Zn**	**Cu**	**In**	**Ga**	**Ge**	**W**	**Ag**
Mean	7,312.5	10,000	2516	59.28	23.2	14.8	195.4	10
SD	324.76	0.00	70.88	2.39	1.17	0.40	6.68	1.10
Variance	105,468.750	0.000	5,024.000	5.706	1.360	0.160	44.640	1.200
Number	4	5	5	5	5	5	5	5

12.3 QUALITY ASSURANCE/QUALITY CONTROL ANALYSIS – 2014 DRILL PROGRAM

This section provides an analysis of the various QA/QC procedures and results employed during the 2014 drill program.

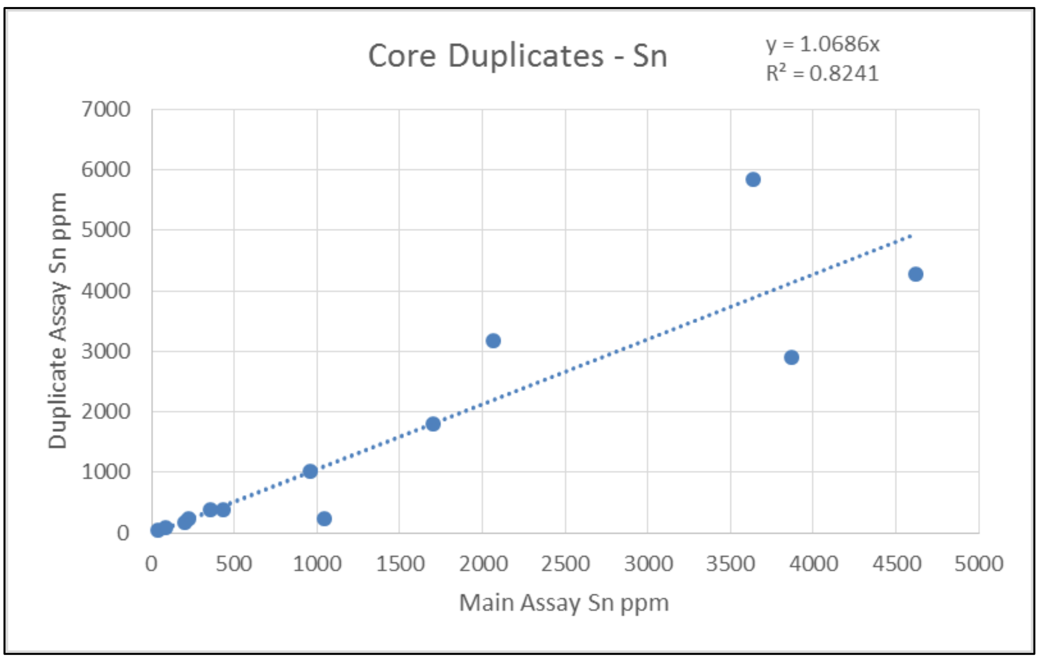
12.3.1 Quarter Core Duplicates

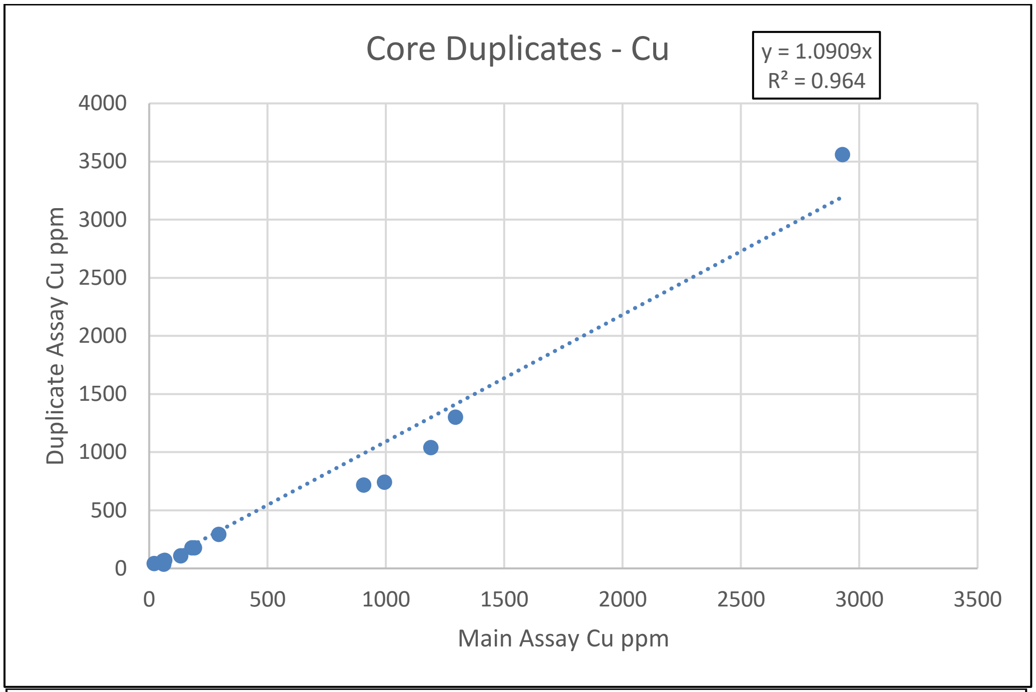
Thirteen ¼-core duplicates were prepared by splitting ½ core. These samples were processed by ALS using the sample preparation and assay procedures described previously. The purpose of the ¼ duplicate was to check on possible bias introduced by core splitting. The results show reasonable correlation but with somewhat high percent relative standard deviation (%RSD) values for Zn, Cu and In. These results are comparable to those obtained from the independent due diligence ¼-core samples. The results show a positive bias for the duplicate compared to the main assay. Overall, the results are consistent with a relatively high level of mineralization heterogeneity over short distances. Table 12.5 and Figure 12.1 summarize the results of the ¼ duplicate assays.

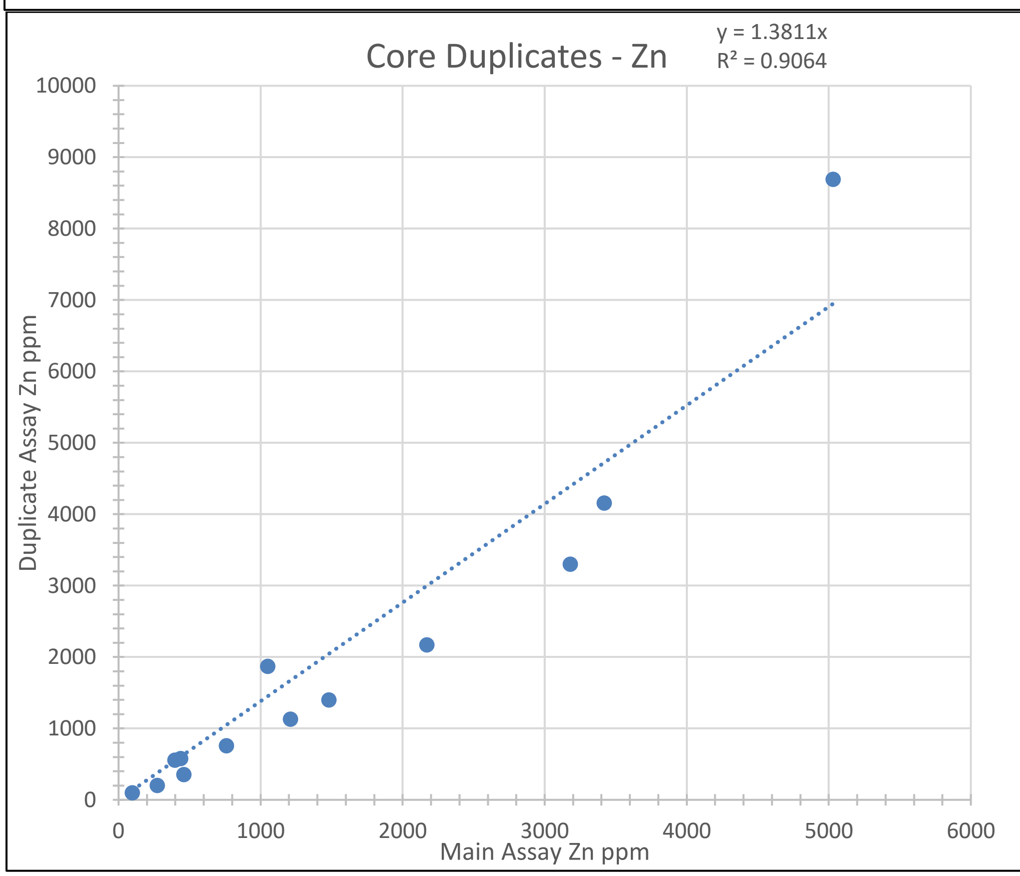
Table 12.5
Quarter Core Duplicate Assay Results – 2014 Drill Program

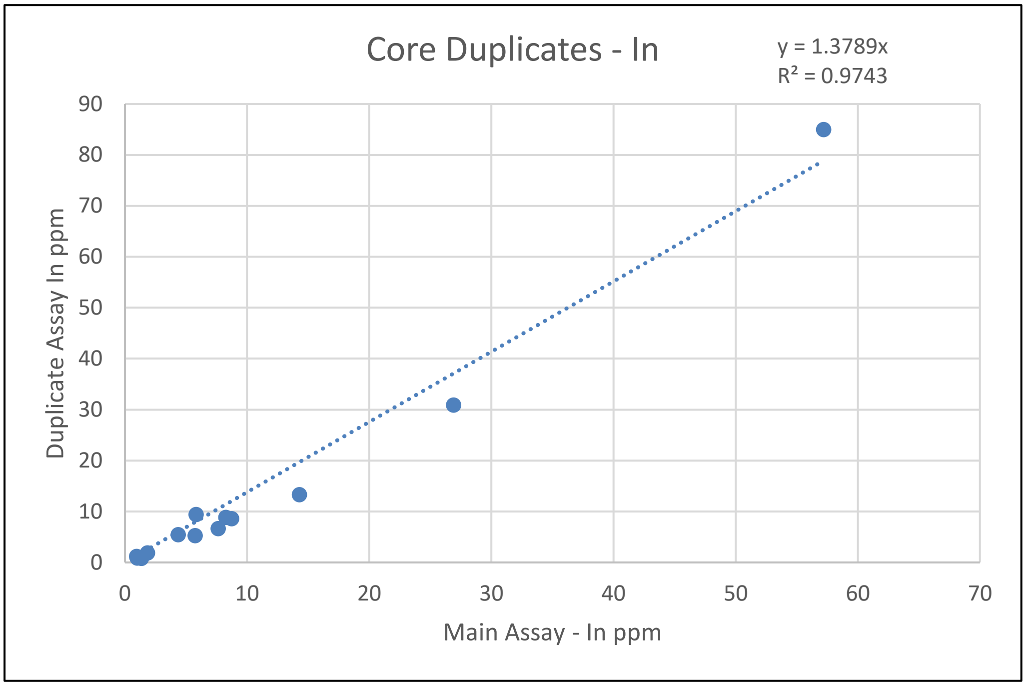
Hole_ID	Main Sample ID	Duplicate Sample ID	Assay Results							
			Main (ppm)				Duplicate (ppm)			
			Sn	Cu	Zn	In	Sn	Cu	Zn	In
EKAV-14-001	R334027	R334028	91	62	273	1.34	78	37	202	0.84
EKAV-14-002	R334067	R334068	1,050	133	459	1.01	220	107	355	0.88
EKAV-14-002	R334088	R334089	2,070	1,190	2,170	14.3	3,170	1,040	2,170	13.35
EKAV-14-003	R334133	R334134	4,620	906	3,420	26.9	4,260	718	4,160	30.9
EKAV-14-003	R334171	R334172	965	192	3,180	8.74	1,010	178	3,300	8.61
EKAV-14-004	R334223	R334224	435	181	1,480	5.76	377	177	1,400	5.29
EKAV-14-004	R334253	R334254	360	21	1,050	5.83	379	41	1,870	9.44
EKAV-14-005	R334282	R334283	224	59	97	1.84	225	63	100	1.89
EKAV-14-005	R334315	R334316	38	66	397	0.95	43	70	556	1.2
EKAV-14-006	R334345	R334346	3,870	995	1,210	7.63	2,880	741	1,130	6.64
EKAV-14-006	R334381	R334382	1,705	1,295	759	8.26	1,780	1,300	761	8.88
EKAV-14-007	R334416	R334417	3,640	2,930	5,030	57.2	5,840	3,560	8,690	85
EKAV-14-007	R334445	R334446	201	294	437	4.36	165	292	580	5.5
Mean			1,482.23	640.31	1,535.54	11.09	1,571.31	640.31	1,944.15	13.72
SD			1,535.59	801.19	1,444.95	14.93	1,820.99	934.45	2,277.30	21.94
% RSD Difference Main vs Duplicate			-12.29%	-20.81%	-23.00%	-25.28%				

Figure 12.1
Quarter Core Duplicate Assays – 2014 Drill Program




Core Duplicates - Cu
y = 1.0909x
R² = 0.964
Duplicate Assay Cu ppm
Main Assay Cu ppm
0
500
1000
1500
2000
2500
3000
3500
4000

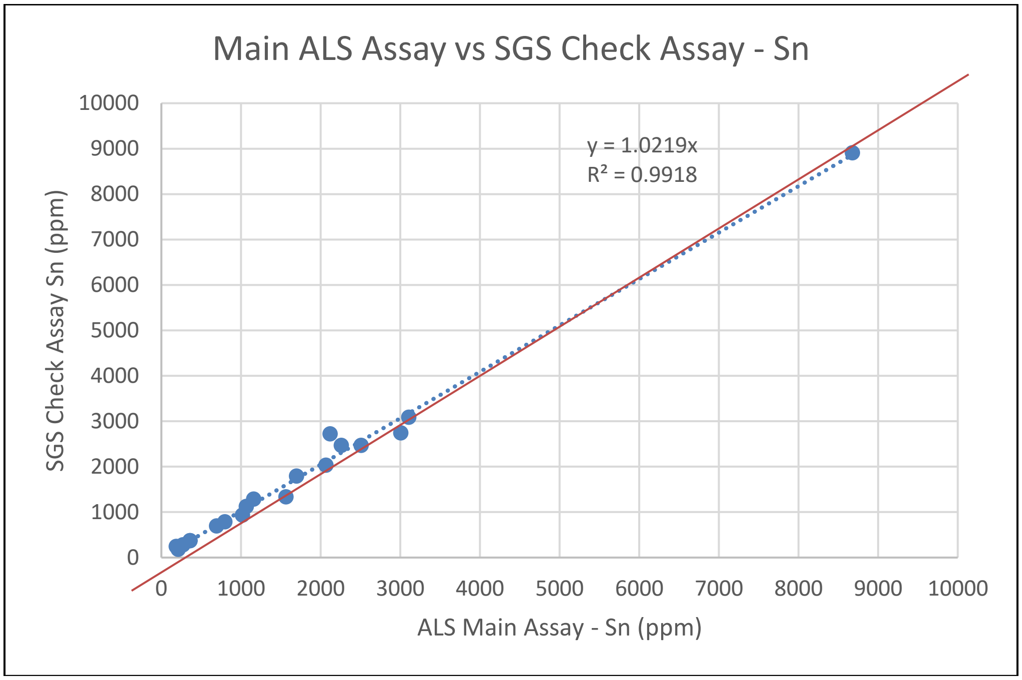

Core Duplicates - Zn
y = 1.3811x
R² = 0.9064
Duplicate Assay Zn ppm
Main Assay Zn ppm
0
1000
2000
3000
4000
5000
6000
7000
8000
9000
10000

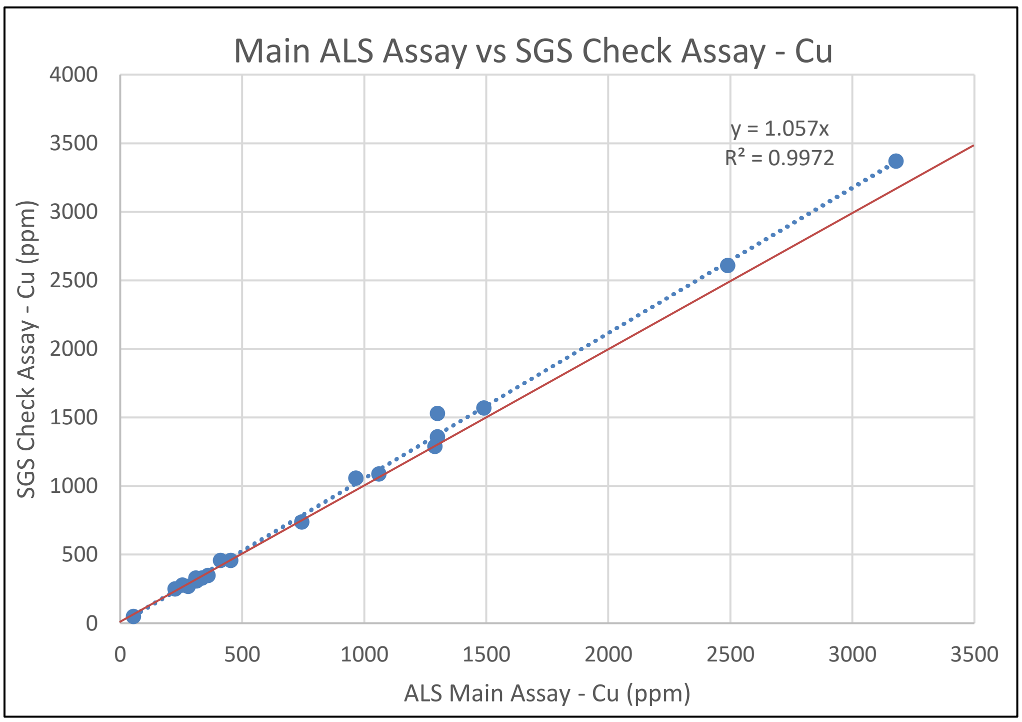


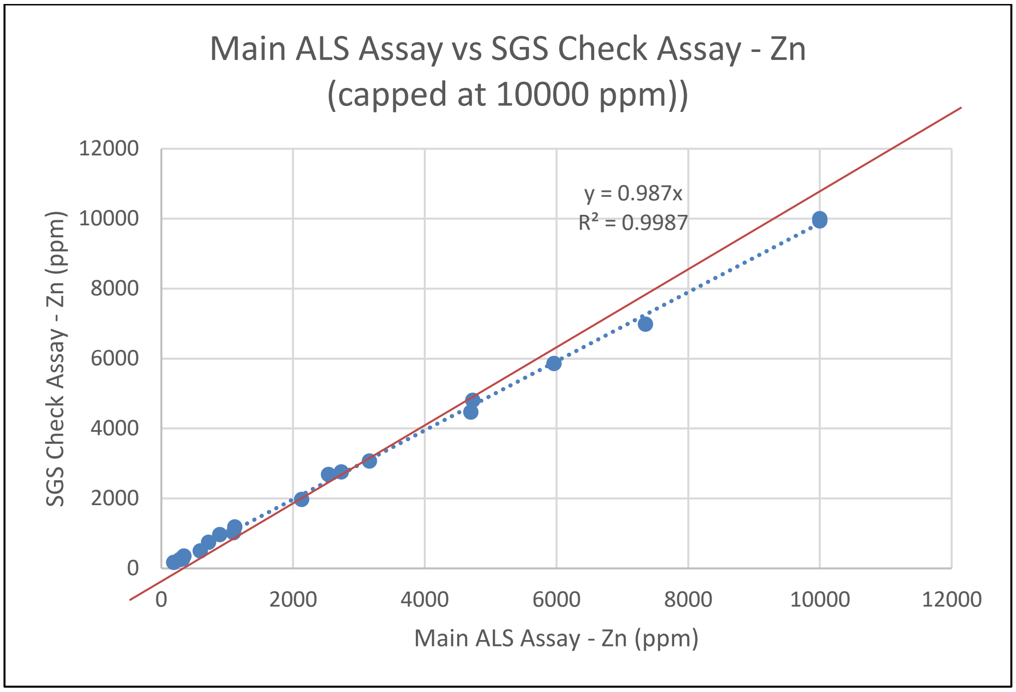
12.3.2 Check Assays

Coarse splits (-10 mesh) for the ½-core samples were prepared and assayed at SGS Lakefield and Actlabs. Comparisons of the assay results provide information on sample preparation and assay method variation. Overall, the results show excellent correlation between the ALS results and those obtained by either SGS Lakefield or Actlabs. Some differences can be noted between the laboratories indicating either a slight positive or negative bias in assays for Sn, Cu, Zn and In, with SGS Lakefield generally being slightly upwardly biased compared to either ALS or Actlabs. Figure 12.2 and Figure 12.3 illustrate the results of the check assays between ALS and either SGS Lakefield or Actlabs. Figure 12.4 provides a comparison between the SGS Lakefield results and the Actlabs results. In all cases, values reporting over limits have been set to 10,000 ppm.

Figure 12.2
Check Assay Results, ALS vs SGS Lakefield






Main ALS Assay vs SGS Check Assay - Zn
(capped at 10000 ppm))
y = 0.987x
R² = 0.9987
SGS Check Assay - Zn (ppm)
Main ALS Assay - Zn (ppm)
0
2000
4000
6000
8000
10000
12000

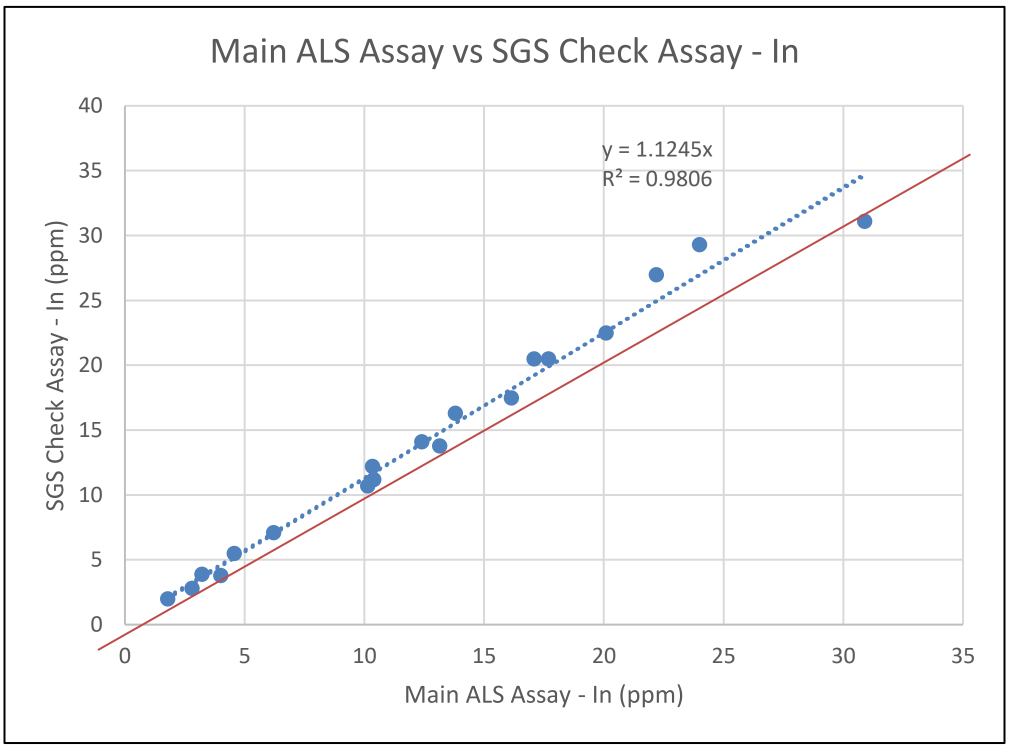
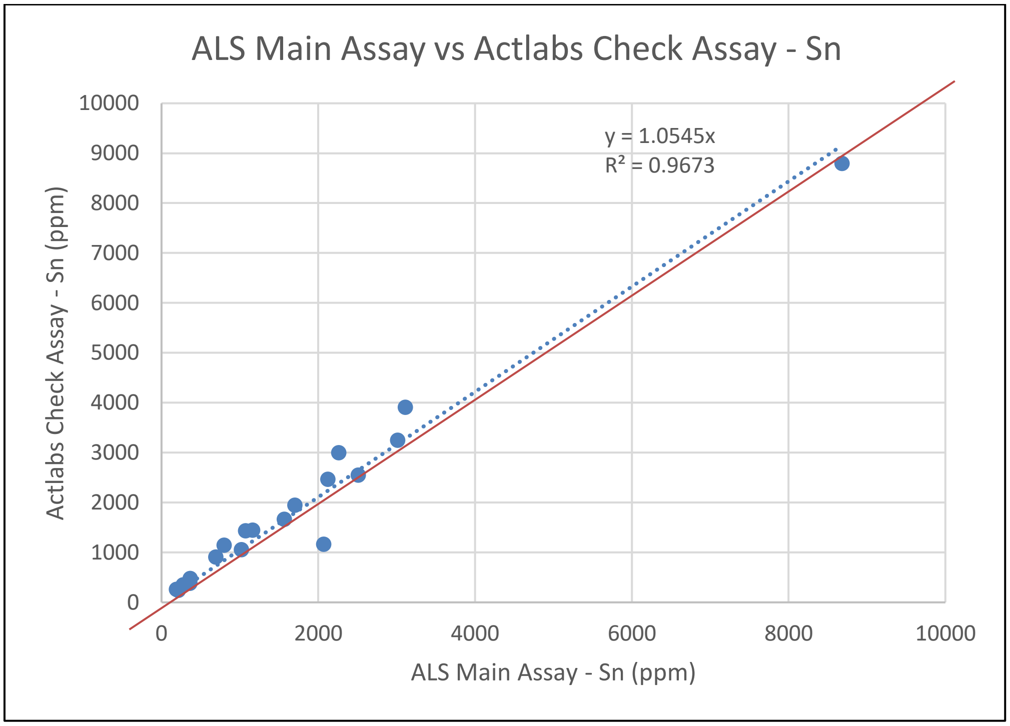
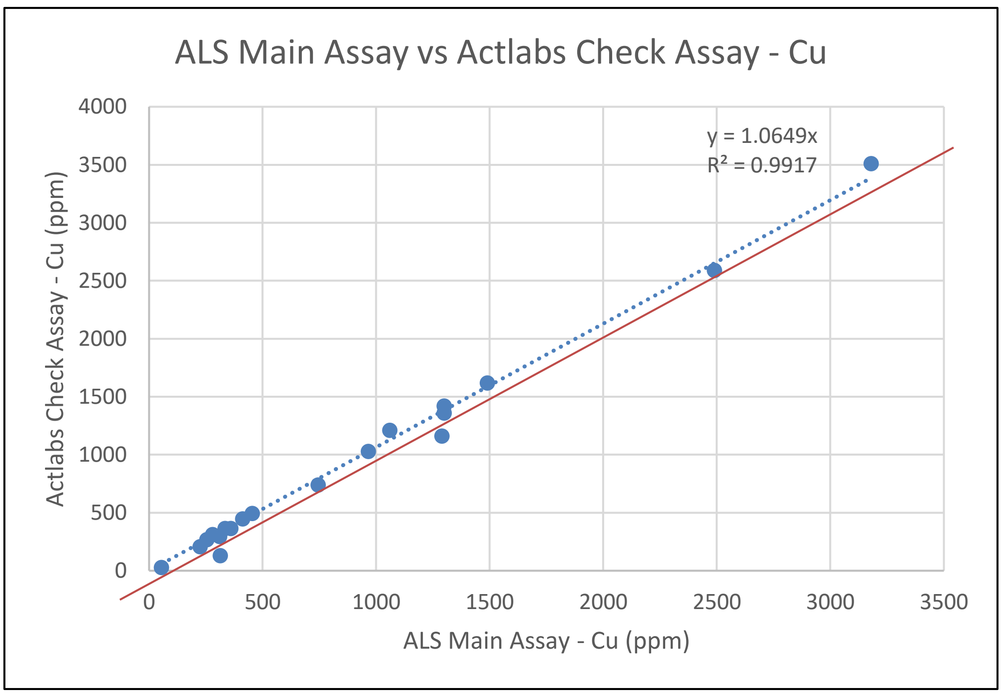
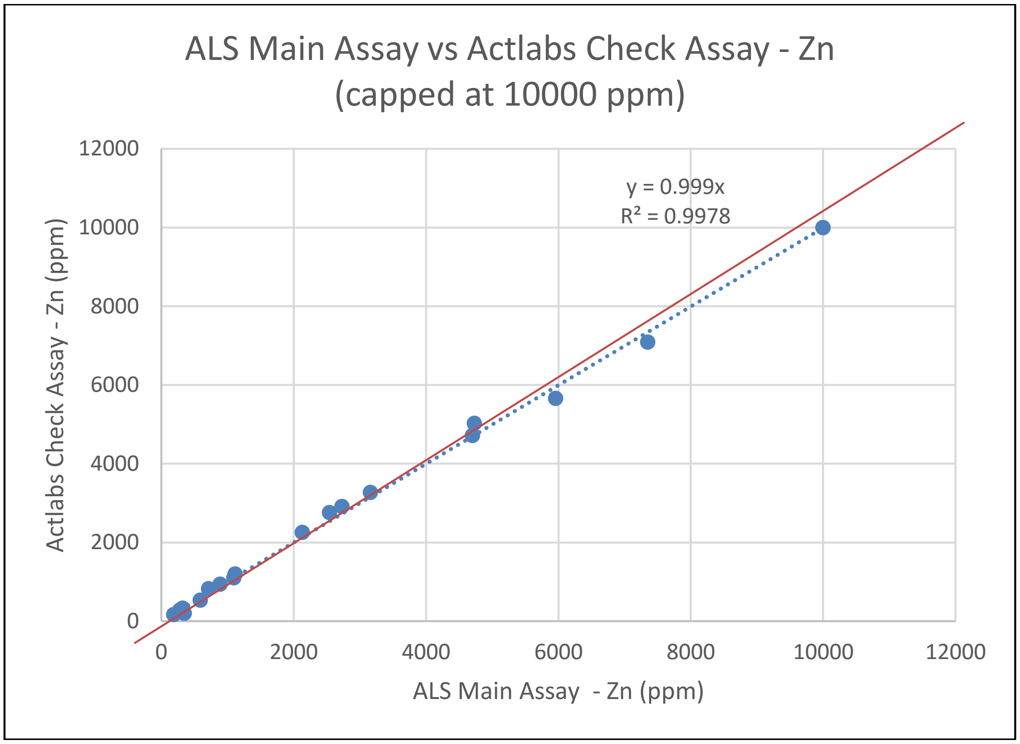

Main ALS Assay vs SGS Check Assay - In
y = 1.1245x
R² = 0.9806
SGS Check Assay - In (ppm)
Main ALS Assay - In (ppm)
0
5
10
15
20
25
30
35
40

Figure 12.3
Check Assay Results, ALS vs Actlabs






ALS Main Assay vs Actlabs Check Assay - Zn
(capped at 10000 ppm)
y = 0.999x
R² = 0.9978
Actlabs Check Assay - Zn (ppm)
ALS Main Assay - Zn (ppm)
0
2000
4000
6000
8000
10000
12000

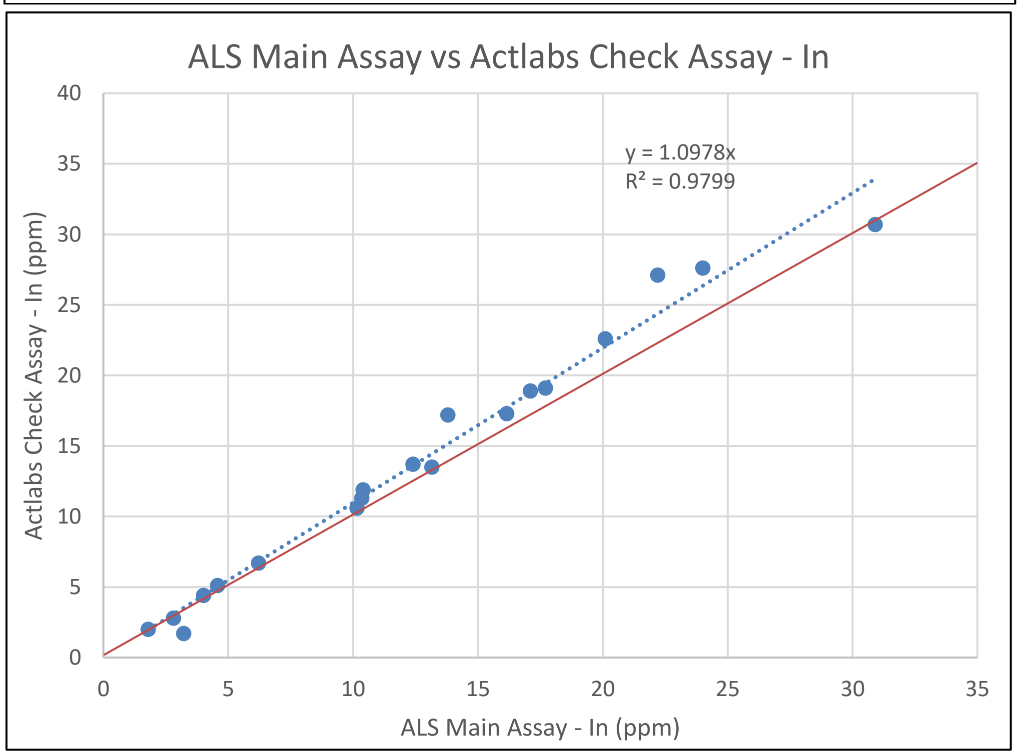
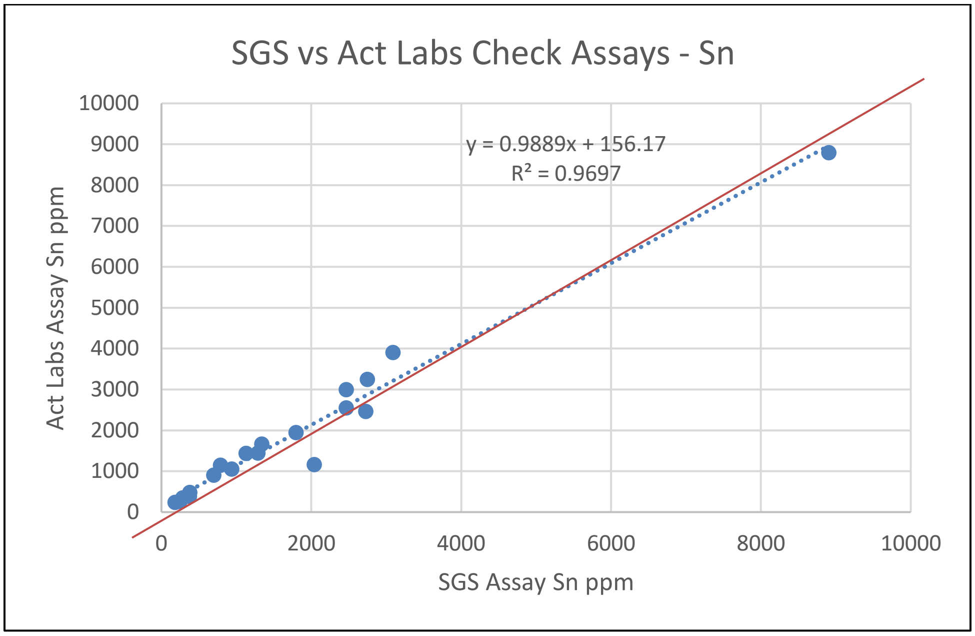
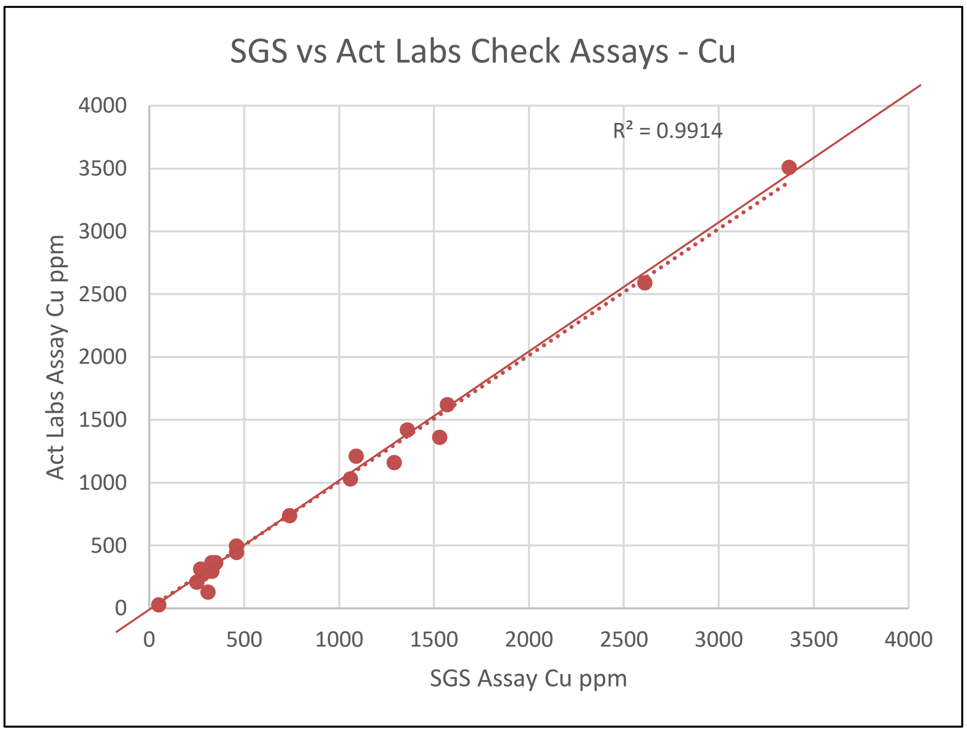
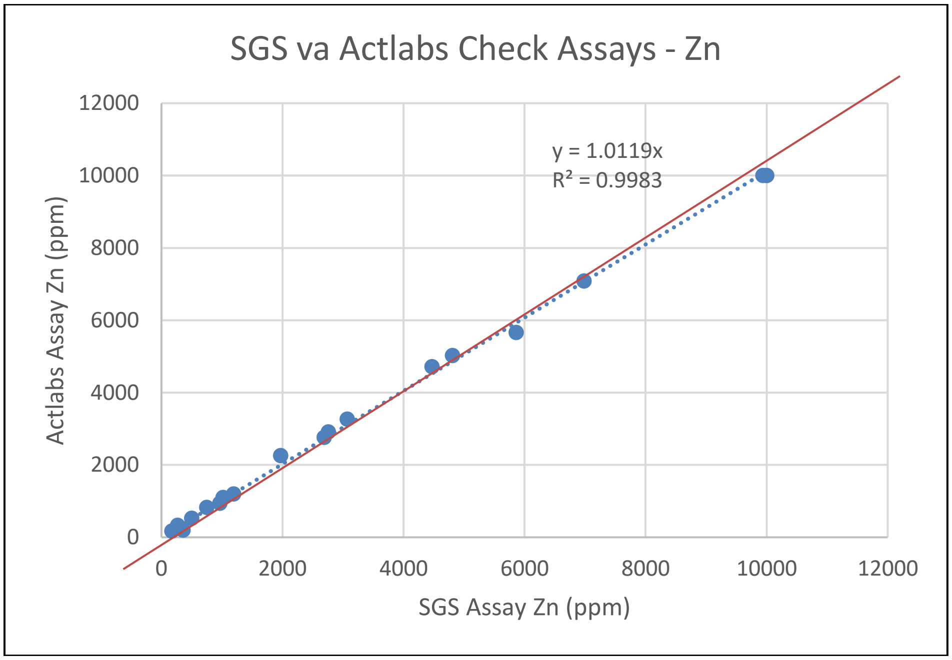
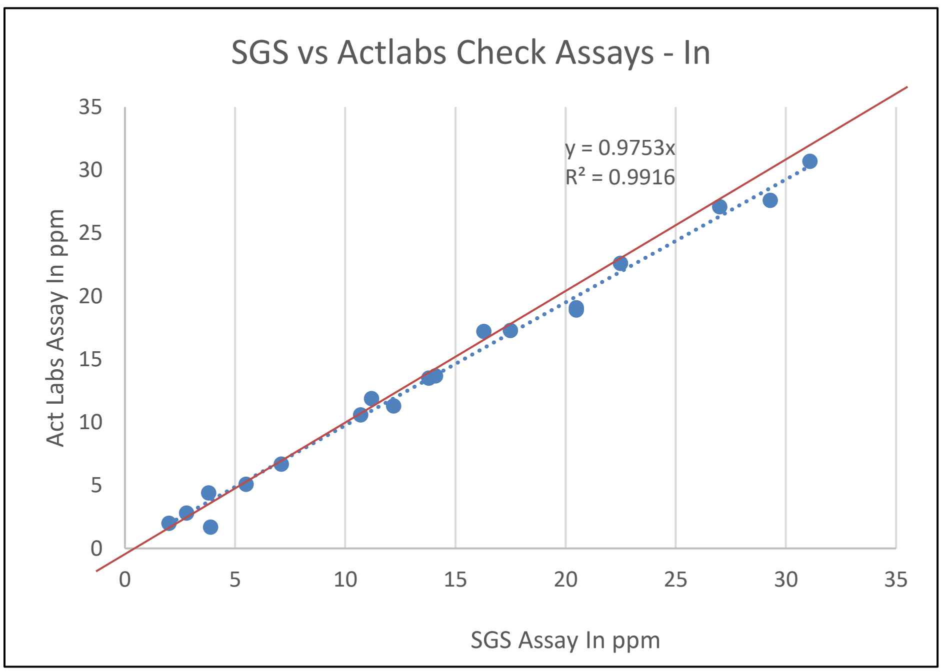

ALS Main Assay vs Actlabs Check Assay - In
y = 1.0978x
R² = 0.9799
Actlabs Check Assay - In (ppm)
ALS Main Assay - In (ppm)
0
5
10
15
20
25
30
35
40

Figure 12.4
Check Assay Results, SGS Lakefield vs Actlabs





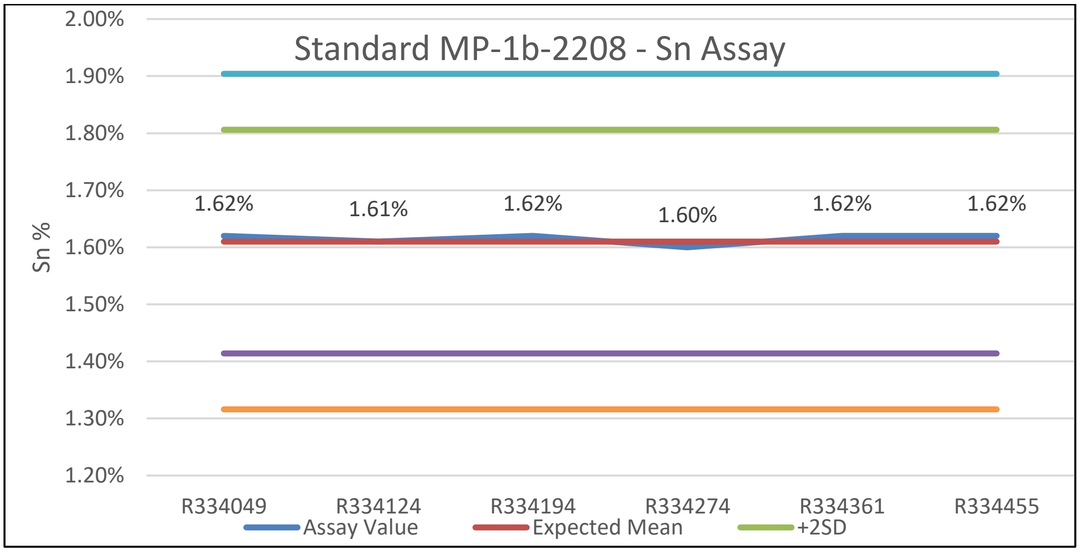


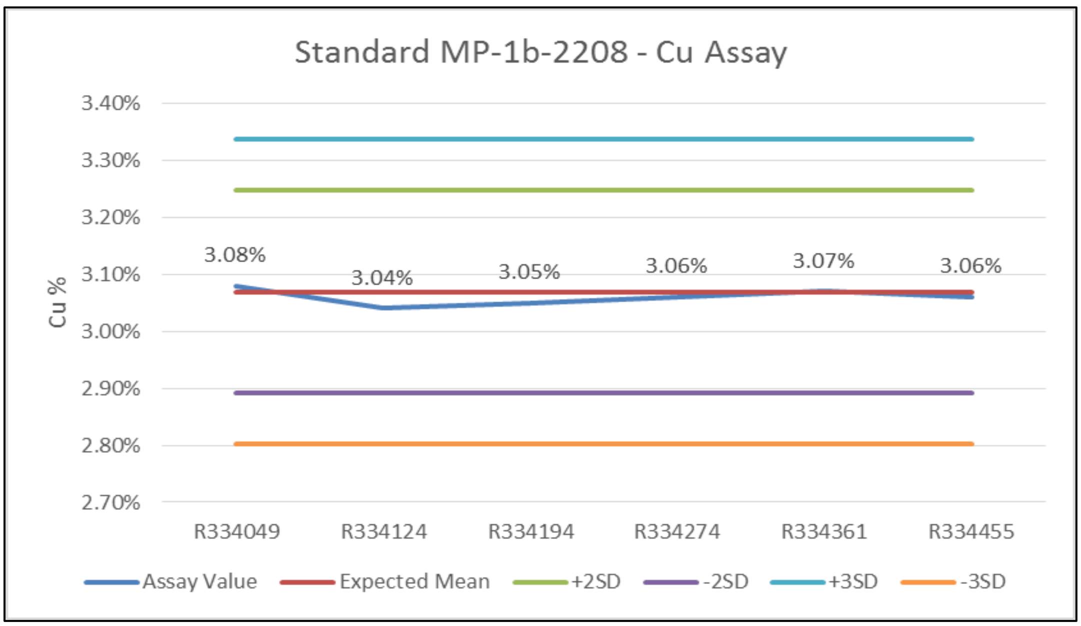


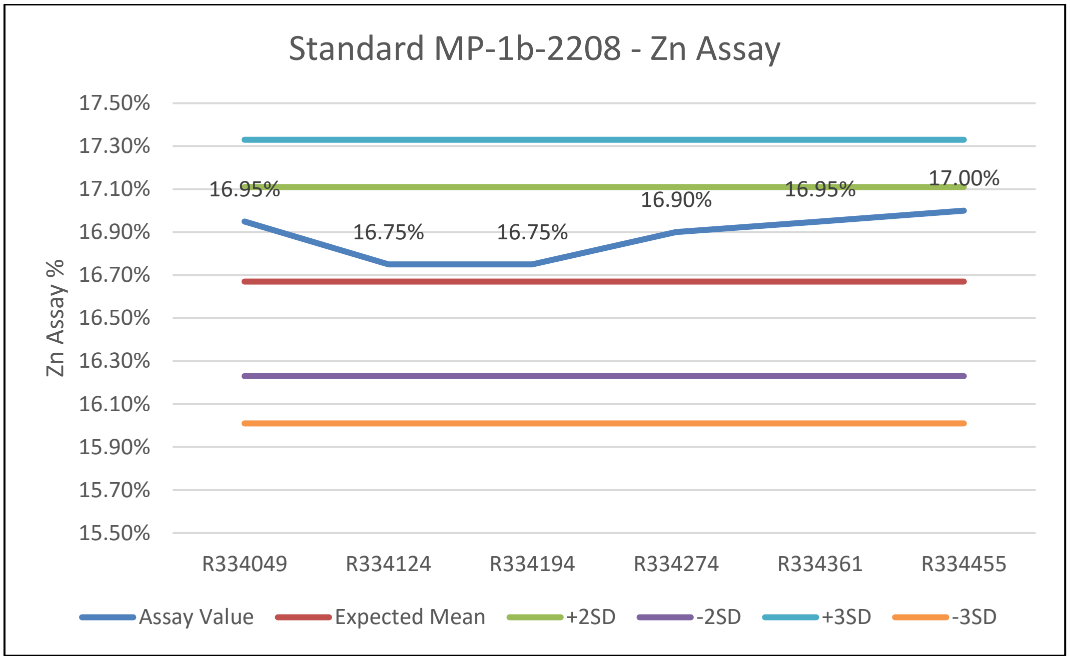
12.3.3 Standards

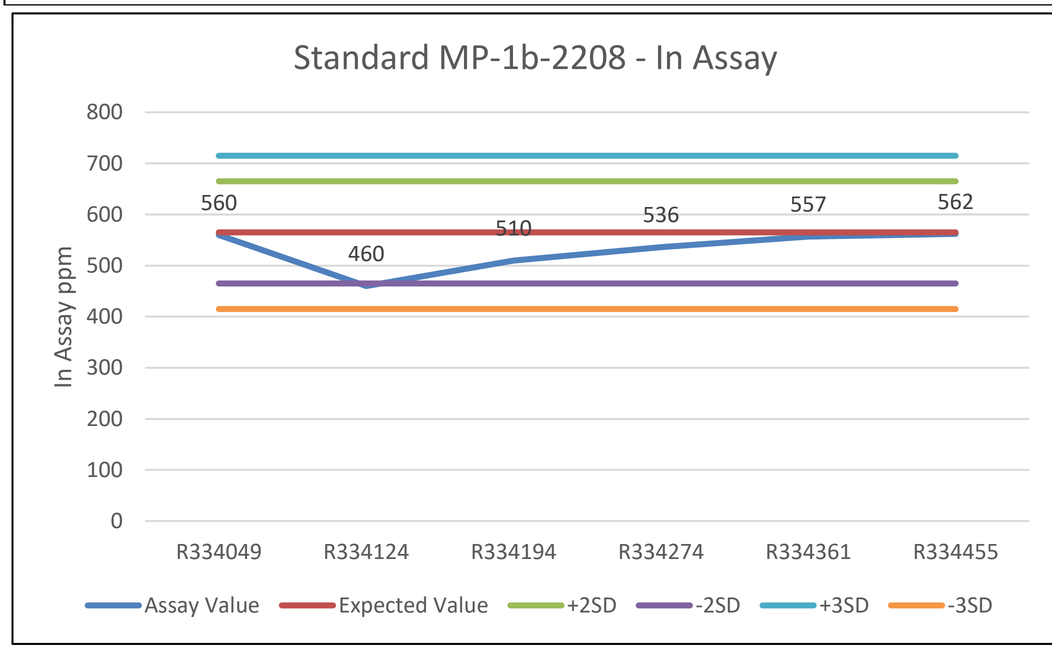
Standards were used to assess laboratory precision. Analysis of the assays of standards used in the main assays at ALS show excellent results, with all values falling within 2 standard deviations of the expected mean. Similar results were observed with the Actlabs and SGS Lakefield assays of check samples. Figure 12.5 illustrates the ALS assay results for standards MP-1b-2208 and two standards obtained from OREAS.

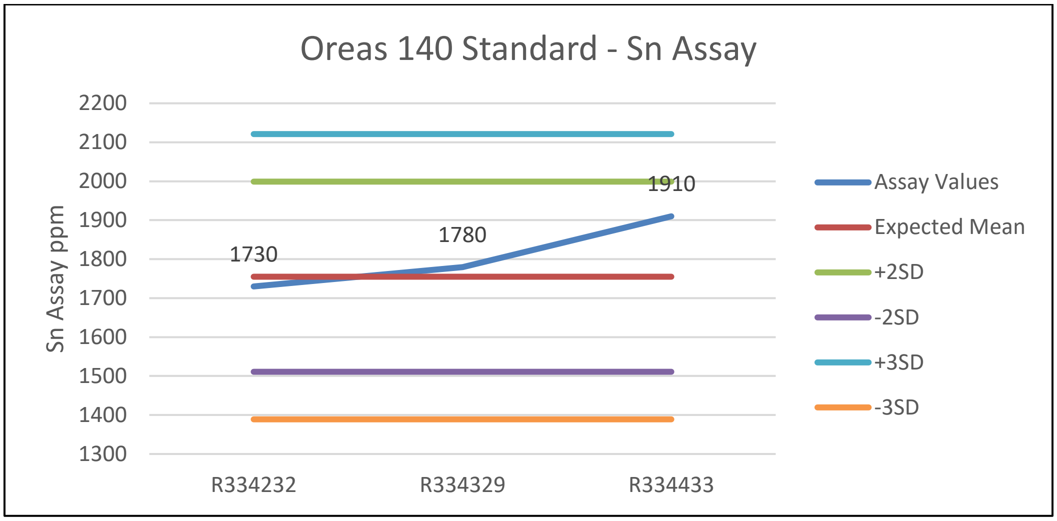
Figure 12.5
Standards Assay Results – ALS

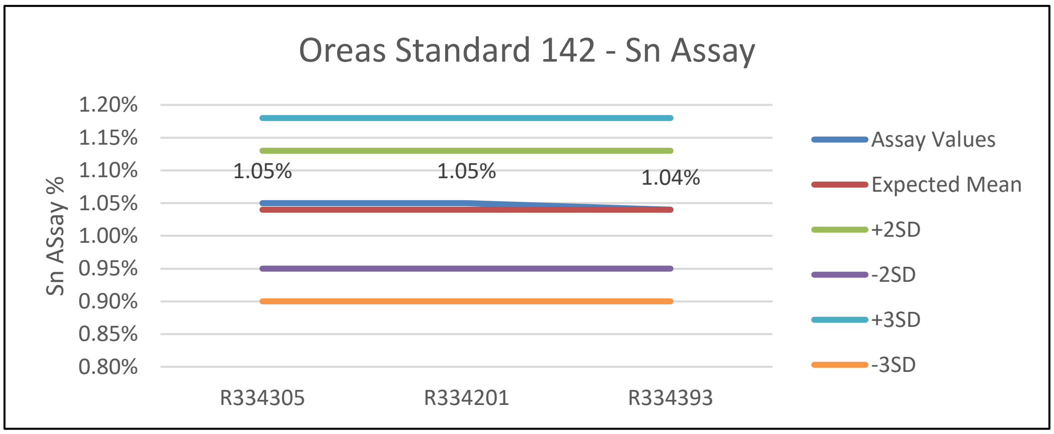





Standard MP-1b-2208 - Zn Assay
17.50%
17.30%
17.10%
16.90%
16.70%
16.50%
16.30%
16.10%
15.90%
15.70%
15.50%
Zn Assay %
16.95%
16.75%
16.75%
16.90%
16.95%
17.00%
R334049
R334124
R334194
R334274
R334361
R334455
Assay Value
Expected Mean
+2SD
-2SD
+3SD
-3SD


Standard MP-1b-2208 - In Assay
800
700
600
500
400
300
200
100
0
In Assay ppm
560
460
510
536
557
562
R334049
R334124
R334194
R334274
R334361
R334455
Assay Value
Expected Value
+2SD
-2SD
+3SD
-3SD





12.3.4 Blanks

See Section 11.1.1 for a description of blank materials. It was previously analysed during a reconnaissance drill program and was known to be low in tin and further evidenced and confirmed by the analyses listed in Table 12.4 from the 2014 drill program.

The results for the laboratory internal blanks used by ALS and Actlabs show no evidence of sample contamination during assaying.

12.4 QA/QC Summary – 2014 Drill Program

The review of the QA/QC data indicates no significant issues with respect to sample preparation, assaying and laboratory precision. There would appear to be minor differences in assay results depending on the method of analysis, with the SGS ICM90A method being slightly upwardly biased compared to either ALS or Actlabs. It is recommended that certified blanks be used to monitor the potential for sample contamination. It is recommended that the insertion rate for standards and blanks be increased to at least 5% of main, duplicate and check samples. It is recommended that both coarse and pulp duplicates and check samples be assayed.

Overall, the author is satisfied that the sample preparation and assay procedures used for the 2014 drill program are sound and that the results can be relied upon for use in resource estimates.

12.5 QUALITY ASSURANCE QUALITY CONTROL 2015 DRILL PROGRAM

12.5.1 QA/QC Protocols

Similar QA/QC protocols were followed in the 2015 drill program as previously used in 2014. Bagged samples were placed in 20 litre plastic pails. The pails were sealed with secure lids and taped closed and the sample number intervals noted on the outside of the pail. Once a sufficient number of samples had been prepared, samples were shipped by courier to Actlabs in Ancaster, Ontario, or the Actlabs sample preparation facility in Fredericton, New Brunswick with the location determined by whether there were backlogs at the New Brunswick preparation laboratory.

The core logging geologist was responsible for making a determination as to whether a sample was potentially mineralized or likely to be waste rock in any open pit. The instructions sent to the laboratory were different according to whether the sample was designated in the first, mineralized, or second, unmineralized, category. The unmineralized potential open pit waste rock was analysed for elements that may indicate potential for Acid Rock Drainage and also for tin to identify any possible missed mineralized intercepts.

Coarse reject check sample splits were shipped by Actlabs to Avalon for insertion of standards and subsequently shipped by Avalon to ALS under the same sample number as the main sample with an identifier letter code. A summary of the number of inserted standards is presented in Table 12.6.

Table 12.6
Inserted Standards Counts

Standard Type	DH Sample Count	Standard Type	Standard Sample Analysis Count	Ratio of QC Standard to DH Samples
Laboratory	2438	40	636	1:4
Avalon	2438	4	203	1:12

The primary laboratory was Actlabs in Ancaster, Ontario, and the secondary laboratory, for duplicate analyses ("Pulp Duplicates" in Table 12.7) was ALS in Vancouver.

Table 12.7
Duplicate Sample Analyses Counts

QC Category	DH Sample Count	QC Sample Count	Ratio of QC Samples to DH Samples
Field duplicate	2438	76	1:32
Lab Pulp Split	2438	49	1:50
Preparation Duplicate	2438	49	1:50
Pulp Duplicate	2438	607	1:4
Check Assay	2438	183	1:13

12.5.2 Core Duplicates

Seventy-six core duplicates were analysed during the 2015 drill program. The results of the Sn analyses are illustrated in Table 12.8 below.

Table 12.8
Core Duplicate Analyses for Tin

	Number	Mean Sn1	Mean Sn2	SD Sn1	SD Sn2	RPHD
ICP	76	1,502	1,567	2,207	2,404	0.82
XRF	36	857	992	1,628	1,986	2.07

Notes to table:
1. The table shows the results of ICP and XRF analyses respectively.
2. Sn1 and Sn2 represent the initial and repeat analysis for tin in ppm respectively.
3. SD is the standard deviation in ppm.
4. RPHD is the relative percentage half difference in percent between Sn1 and Sn2.

The data in the table and the regression plot in Figure 12.6 and Figure 12.7 illustrate that the duplicate core analyses gave acceptably similar results implying a lack of significant bias in the core sampling.

Results on other elements for 76 duplicates all analysed by ICP indicate a relative percentage half difference for zinc, copper, indium and silver of 3.6%, 3.1%, -1.7% and 9% respectively. These are considered acceptable differences.

Figure 12.6
Scatterplot of 1st and 2nd Halves of Drill Core Analyses for Duplicate Core Samples (ICP for Sn)

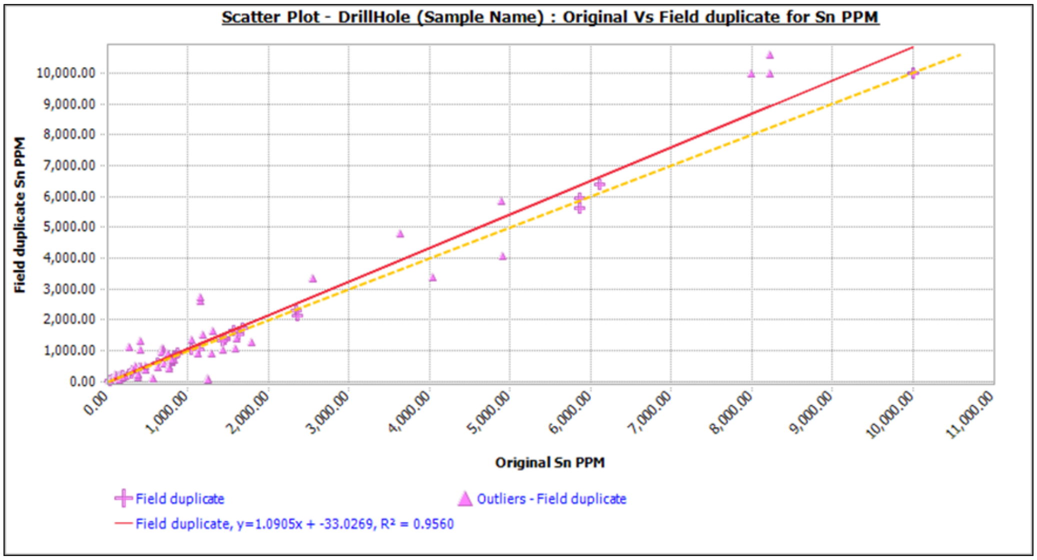
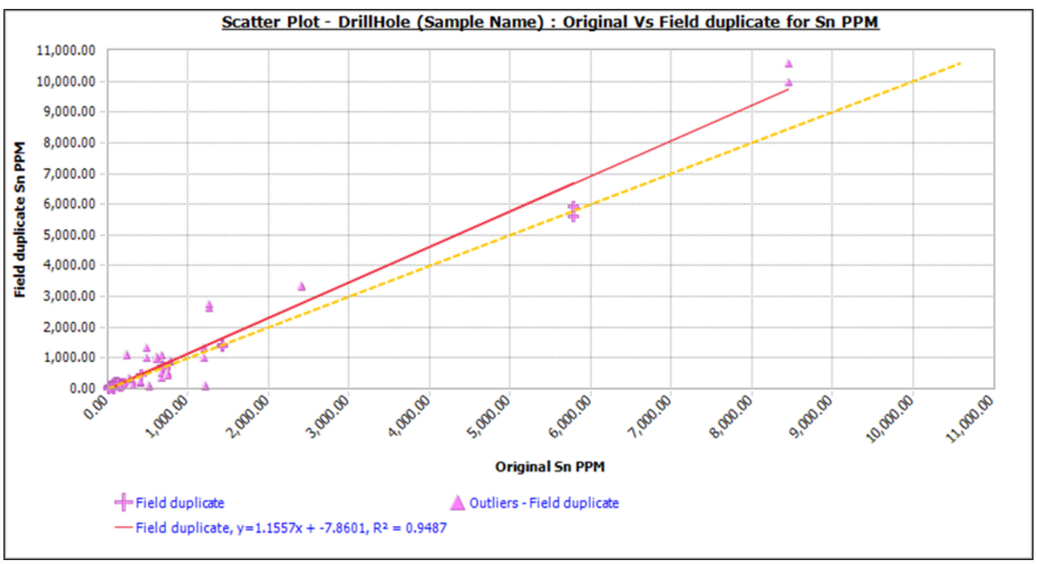


Figure 12.7
Scatterplot of 1st and 2nd Halves of Drill Core Analyses for Duplicate Core Samples (XRF for Sn)



Notes to figures:

1. Original Sn ppm corresponds to Sn1 in Table 12.8 and Field Duplicate Sn ppm corresponds to Sn2
2. Orange dashed line is ideal line for Sn1=Sn2 and red line is regression line for actual data.

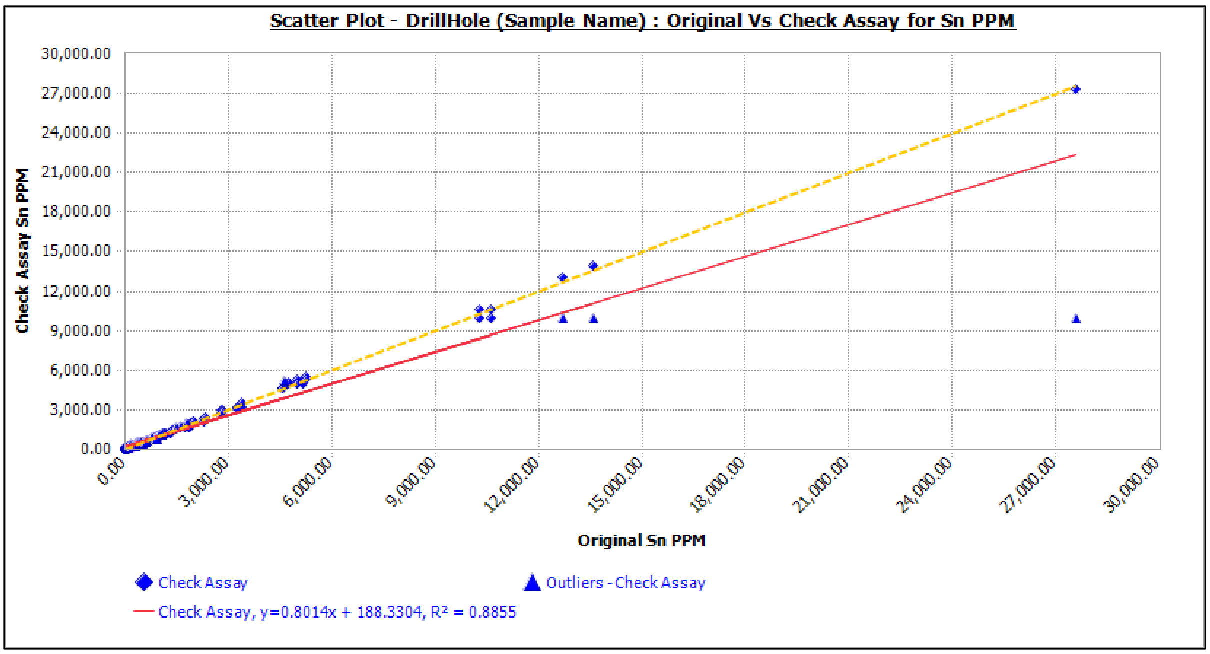
12.5.3 Check Assays

As noted in Table 12.7, 183 check analyses were completed, which is an average ratio of a check analysis every 13 analyses. The results for tin are illustrated in Table 12.9 and Figure 12.8 and demonstrate close agreement between the first laboratory, Actlabs, and the second laboratory, ALS. There is a relative half difference of -1.2% for ICP with ALS slightly higher and a difference of -7.9% for XRF again with ALS higher than Actlabs. This implies that use of the data from Actlabs will be slightly conservative in terms of tin levels which is a similar conclusion as for standard analyses.

Table 12.9
Comparison of Actlabs (Sn1) and ALS (Sn2) on Pulp Duplicates

Method	No. of Samples	Mean Sn1 ppm	Mean Sn2 ppm	SD Sn1	SD Sn2	CV Sn1	CV Sn2	sRPHD
ICP	183	1036	1097	1841	2013	1.78	1.84	-1.18
XRF	151	1224	1261	2997	3011	2.45	2.39	-7.87

Figure 12.8
Regression Plot of Pulp Duplicates from Actlabs (Original Sn ppm) and ALS (Check Assay Sn ppm)



12.5.4 Standards

Four standards were routinely used for checks on the laboratory batches being standard LG, AG, HG and VG. The Sn values are given in Table 12.10 and run charts for the standard analyses in Figure 12.9 to Figure 12.12.

The results show a consistent negative bias in both Table 12.10 and the figures for both ICP and XRF analyses between the certified value from the analytical Round Robin and the 2015 drill program results.

It is also clear that the negative bias for ICP analyses is higher at high grades as standard EK-HG1 has a negative bias of 7.6%. The analyses in the 2015 drill program on the high-grade standard are 7.6% below the Round Robin average of acceptable laboratories. For XRF, this trend of greater negative bias at higher tin values is not apparent and in fact the negative bias lessens at higher tin values.

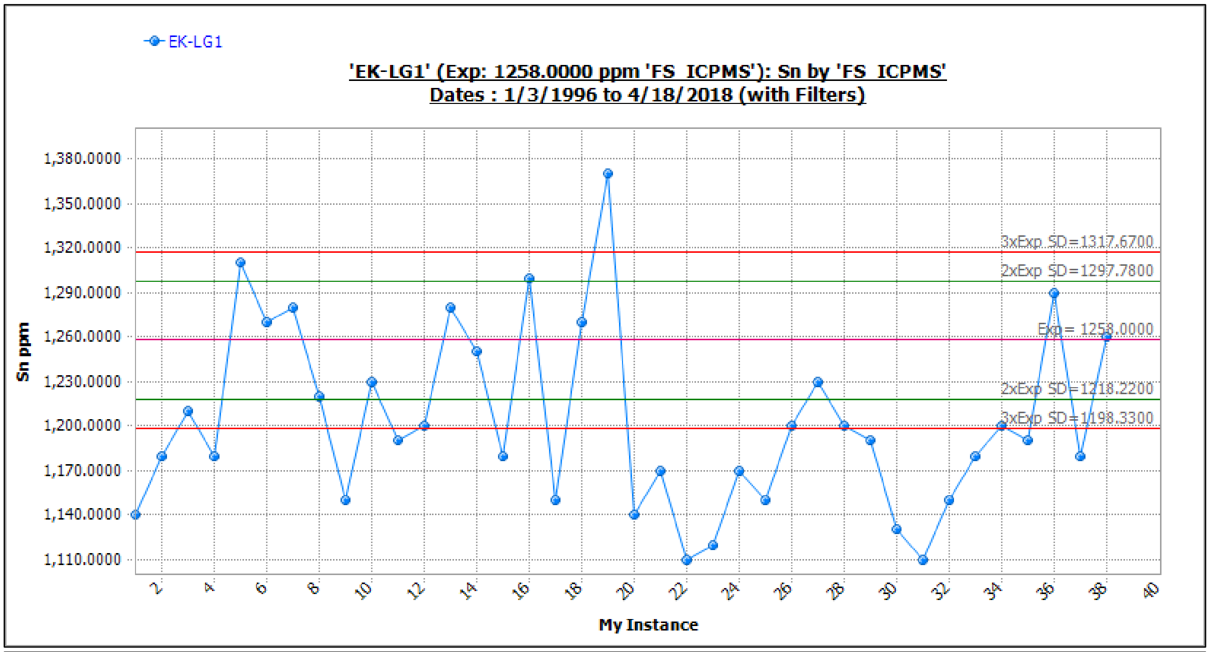
The overall bias values are considered acceptable and suggests any results might be conservative rather than overestimated. Also, high tin values were consistently reassayed utilizing XRF which shows a negative bias of 3.5% at high tin levels in standard VG1.

An additional conclusion from this data is that it demonstrates the overall superior results for XRF analysis compared to ICP for Sn.

Table 12.10
Standard Analyses for Tin from 2015 Drill Program Compared to Certified Values for Standards

Sn Standard(s)				No. of Samples	2015 Drill Program			
Std Code	Method	Certified Value ppm Sn	Certified SD		Mean ppm Sn	SD	CV	Mean Bias
EK-LG1	FS_ICPMS	1,258	19.9	38	1,203	61.17	5.1%	-4.3%
EK-AG1	FS_ICPMS	1,828	47.1	50	1,740	102.58	5.9%	-4.8%
EK-HG1	FS_ICPMS	9,391	108.1	28	8,682	380.08	4.4%	-7.6%
Sn Standard(s)				No. of Samples	2015 Drill Program			
Std Code	Method	Certified Value ppm Sn	Certified SD		Mean ppm Sn	SD	CV	Mean Bias
EK-LG1	XRFFS	1,273	37.5	41	1,185	31.85	2.7%	-6.9%
EK-AG1	XRFFS	1,845	45.7	47	1,736	49.48	2.9%	-5.9%
EK-HG1	XRFFS	9,308	222.8	43	8,909	171.79	1.9%	-4.3%
EK-VG1	XRFFS	12,920	393.2	35	12,474	167.44	1.3%	-3.5%

Figure 12.9
Run Chart for Tin Analysis of Standard EK-LG1 (ICP and XRF)



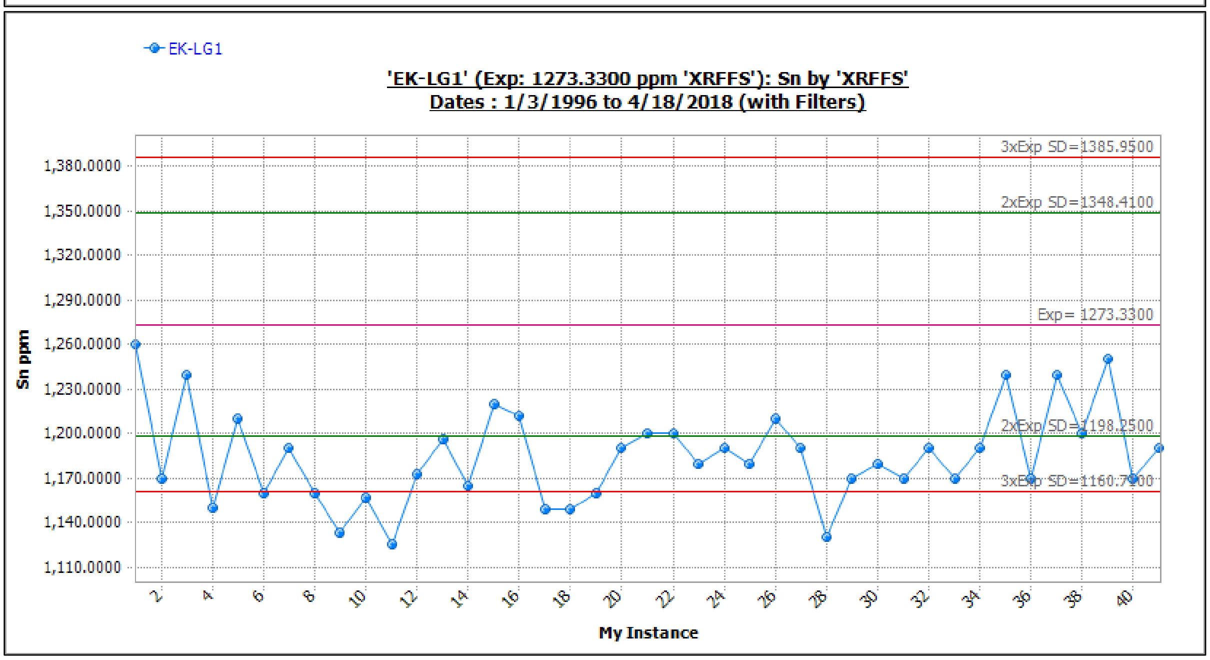
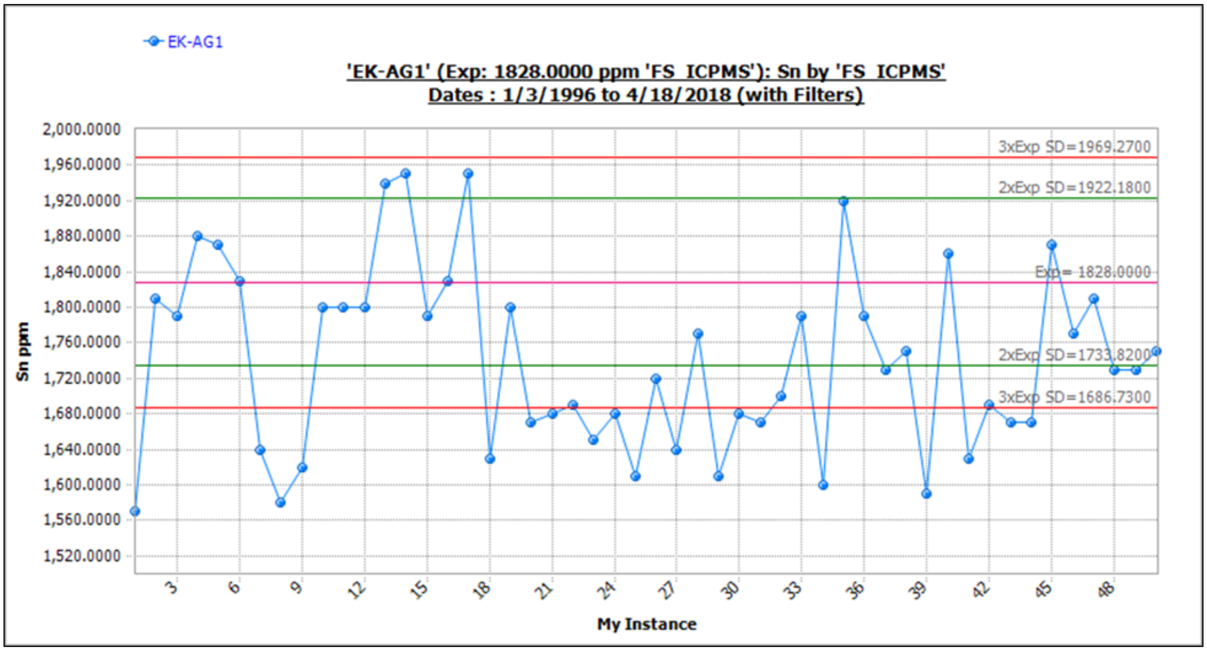


Figure 12.10
Run Chart for Tin Analysis of Standard EK-AG1 (ICP and XRF)



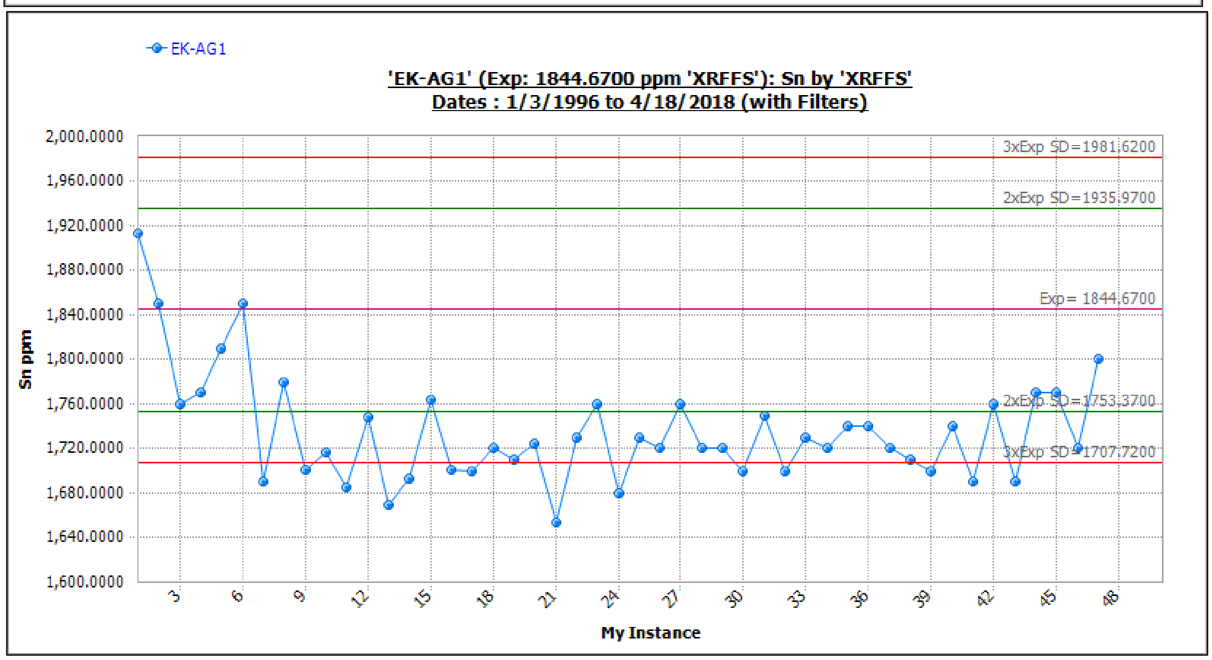
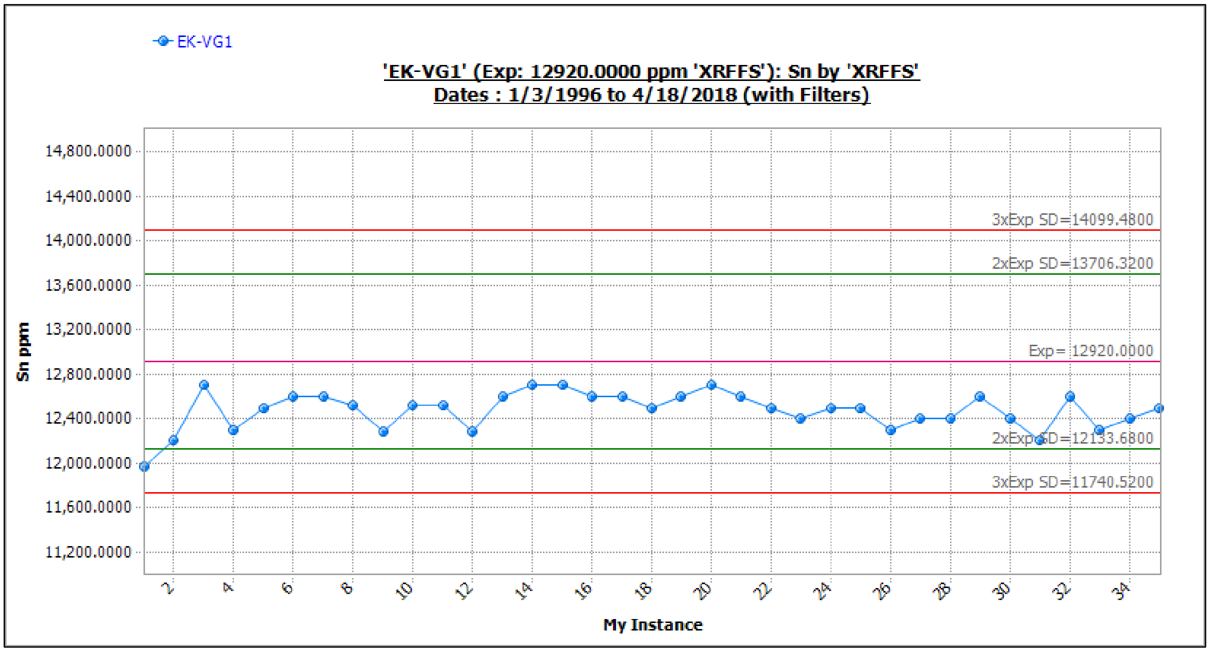


Figure 12.11
Run Chart for Tin Analysis of Standard EK-HG1 (ICP and XRF)

Figure 12.12
Run Chart for Tin Analysis of Standard EK-VG1 (XRF)



Similar data is available for other elements. Zinc data, as this is the more abundant of the other base metals in the deposit, is given in Table 12.11. As can be seen, the zinc values are in excellent agreement with the certified values and bias varies between positive 1.8% and negative 0.6%.

Table 12.11
Standard Analyses for Zinc from 2015 Drill Program Compared to Certified Values for Standards

Standard	Method	Certified			2015 Drill Program				
		Exp Method	Exp Value	Exp SD	Number of Samples	Mean Zn	SD	CV	Mean Bias
EK-LG1	FS_ICPMS	FS_ICPMS	2,728	159	45	2,777	175	0.06	1.80%
EK-AG1	FS_ICPMS	FS_ICPMS	2,868	203	52	2,917	187	0.06	1.68%
EK-HG1	FS_ICPMS	FS_ICPMS	3,946	218	46	3,924	264	0.07	-0.57%

Results not illustrated are those of Cu, In and Ag, but these show biases in the range of negative 1.6 to positive 4.8%, negative 7% to positive 5% and positive 1.4% to positive 3.9%, respectively. These values are considered acceptable especially considering the low economic contribution of these elements to the value of the deposit.

12.6 QA/QC Summary – 2015 Drill Program

In conclusion, a program of QA/QC was conducted in the 2015 drill program that included analysis at Actlabs, Ancaster, Ontario, of standards, blanks, duplicate core samples and duplicate pulp analyses with these latter analyses conducted at ALS in Vancouver, British Columbia.

The results on the standards and duplicates suggest that Actlabs may have a slight negative bias in analyses of the order of 4% to 8% in tin. Bias for other elements including Zn, Cu, In and Ag are variable, being both negative and positive, and of lower magnitude. As all biases present are indicated at levels below 10% and in most cases less than 5%, the analytical data is considered acceptable for resource estimation.

12.7 Historic Shell and RAL Sampling Programs

This section details the historic sampling methodologies employed during the exploration and development of the East Kemptville deposit, and the due diligence sampling programs undertaken by Avalon to validate the historic sampling data.

The independent laboratories used by Shell and RAL included Atlantic Analytical Laboratory, X-Ray Assay Laboratory (XRAL) and Bondar Clegg.

No records were found regarding Atlantic Analytical Laboratory in a search of the NRCan Laboratories list.

The laboratory referred to as XRAL (X-Ray Assay Laboratory) in the Shell and RAL literature such as the Feasibility Study of East Kemptville was acquired by the SGS Group in 1988 and in effect became SGS's Canadian Laboratory (reference online document at https://www.sgs.ca/-/media/local/canada/documents/flyers-and-leaflets/sgs-min-lab-geochemical-services-north-america-flyer-en-11.pdf).

Bondar Clegg laboratory operated from 1962 and was taken over by ALS Laboratories in 2001.

There is no evidence that the three laboratories, XRAL, Bondar Clegg and Atlantic Analytical Laboratory, had any relationship with Shell or Rio Algom, except as independent assay laboratories. The authors used XRAL and Bondar Clegg at the time of the East Kemptville exploration and knew them as independent laboratories with no formal relationship (ownership or management) to any operating mining company.

The exploration undertaken by Shell and RAL is described in Section 6.0.

Table 12.12 summarizes the diamond drill core sampling procedures, sample preparation methods and analytical techniques employed by Shell and RAL from 1978 through to February, 1983.

Drill core recovery for both Shell and RAL was reported as very high (>95%).

Table 12.12
Summary of Shell and RAL Drill Core Sampling, Sample Preparation and Analytical Techniques

Drilling Period	Sample Numbers	Sample Preparation	Analytical Procedures
Mar 1978-Apr 1979	0001-0648	BQ core split to varying length, typically 3 m. Early holes typically sampled at 1 m intervals. Max sample length 7 m., shortest <0.5 m. All core sampled.	Crush to -¼-in, riffle out 50 – 250 g split, crush to -200 mesh, split 5 g for assay. Main assay laboratory, Atlantic Analytical. Sn assayed to 0.10% by wet chemical method developed by CANMET. Geochemical assays on samples <0.10% Sn by XRAL and Bondar Clegg using XRF.
Sep 1979-Dec 1979	1000-3400	BQ core split to 3 m sample length. All drill core sampled.	As above
Jun 1980-Sep 1980	4000-6000	BQ core split to 3 m sample length. All drill core sampled.	Sample crushed to -18 mesh, 250 g riffle split, pulverize to -200 mesh, 5 g split for assay. Analysis at XRAL by XRF.
Nov 1982-Feb 1983	10501-10718 and 11211-11500	BQ core split to 3 m sample length. All drill core sampled.	Total sample crushed to -¼-in, one half split, crushed to -20 mesh, riffle out 500 g, pulverize to -200 mesh, 5 g split for assay. All sample preparation and analytical work at XRAL.

Source: RAL, 1983.

Details of the analytical procedures employed during the Shell and RAL exploration programs are noted below.

12.7.1 Atlantic Analytical Services Ltd.

Sample preparation and analytical procedures employed by Atlantic Analytical Services Ltd. (Atlantic Analytical) for Shell samples were:

- Sn Assay: Analytical methods based on “Methods for the analysis of ores, rocks and related minerals”, E.M. Donaldson, ed., Mine Branch Monograph 881, Inorganic Analytical Research Section, Mines Branch, Department of Energy, Mines and Resources, Ottawa.
- Tin assay by titremetic method based on sodium peroxide-sodium carbonate fusion, HCL dissolution and ammonium hydroxide precipitation followed by SbCl3 reduction and potassium iodate titration. Standardize against known Sn standard prepared in same manner.

$$\% Sn = \frac{(V_s - V_b) \times Sn(eq) \times 100}{\text{Sample wt (mg)}}$$

$$\text{Where } Sn(eq) = \frac{\text{wt Sn in aliquot taken (mg)}}{V_s - V_b}$$

and V_s = Vol (ml) % of potassium iodate solution required by Sn
V_b = Vol (ml) of potassium iodate solution required by blank

12.7.2 X-Ray Assay Laboratories

Sample preparation and analytical procedures employed by X-Ray Assay Laboratories (XRAL) for Shell samples were:

Sample Preparation:

Jaw crush and cone crush total sample to -¼-in. Pulverize total sample to -100 mesh. Riffle out 250 gm aliquot and mill in ring mill to -200 mesh

Analysis:

Sn – initial XRF assay on loose powder. Calibrate using 1,000 ppm synthetic standard. Accept values <500 ppm as final. For >500 ppm samples, pressed pellet XRF method (8 gm, 32 mm dia.) using Sb as an internal standard. Calibrate instrument using synthetic standard for >0.1% Sn <5%. CANMET standard KC-1 used as control standard, plus some previously analysed Shell samples. 10% of samples run as check samples (repeats).

Cu and Zn– mixed acid dissolution followed by AA. Not all samples analysed for Cu and Zn

Sample preparation and analytical methods employed by XRAL for RAL samples were:

Sample Preparation:

For BQ core: 2-4 kg sample, crush to -20 mesh, riffle out 500 gm, pulverize to -200 mesh.

For HQ core: ~25 kg core, crush to -¼-in, split (save rejects), crush to -20 mesh, riffle out 500 g (save rejects), pulverize to -200 mesh.

Analysis:

Sn– XRF using PW1410 instrument. Initial geochemical assay based on loose powder XRF analysis of -200 mesh material. Reported detection limit of 20 to 1,000 ppm. Values above 1,000 ppm assayed using pressed pellet method incorporating internal Sb standard. Pellet produced using 4 g silica and fixed amount of internal Sb standard. Method reported valid up to 5% Sn. CANMET Sn standard inserted in each batch. 10% of samples re-assayed.

12.7.3 Bondar Clegg

Assay methods used by Bondar Clegg for Shell samples included the following:

Atomic Absorption (AA) after hot Lafort aqua regia digestion
Cu, Pb, Zn, Co, Ni, Fe, Mn, Mo, Ag, Cd
Background correction applied for Pb, Co, Ni, Ag, Cd

XRF (total metal), pressed pellet method, Siemans SRS WDX spectrometer.
Ba, Cr, Nb, Rb, Sr, Th, Ti, Sn, V, Zr

Specific Methods:

W – carbonate flux fusion, water leach, colorimetric assay with zinc dithiol.

U – hot nitric acid extraction, sodium fluoride fusion, fluorometer.

As – nitric/perchloric acid extraction, colorimetric assay by hydride generation.

Sn – by iodide fusion using ammonium iodide and HCl leach, flame AA assay.

Ca, Mg, Na, K – extracted with nitric/perchloric/HF mixture, flame AA assay.

F – sodium carbonate/potassium nitrate fusion, hot leach, pH buffer, specific ion electrode.

Hg– nitric/HCl acid digestion in K_2MnO_4, mercury reduction with stannous sulphate, flameless AA.

Au – fire assay/flame AA or carbon rod/flameless AA.

The classical wet chemical assay methods noted above all require skilled chemists to achieve acceptable precision and accuracy and are subject to a wide range of experimental error. Accordingly, the precision of the methods tends to be lower and the limits of detection higher than the instrumental methods currently used. However, inter-laboratory comparisons made at the same time using the methods may be expected to show similar variances in precision and accuracy.

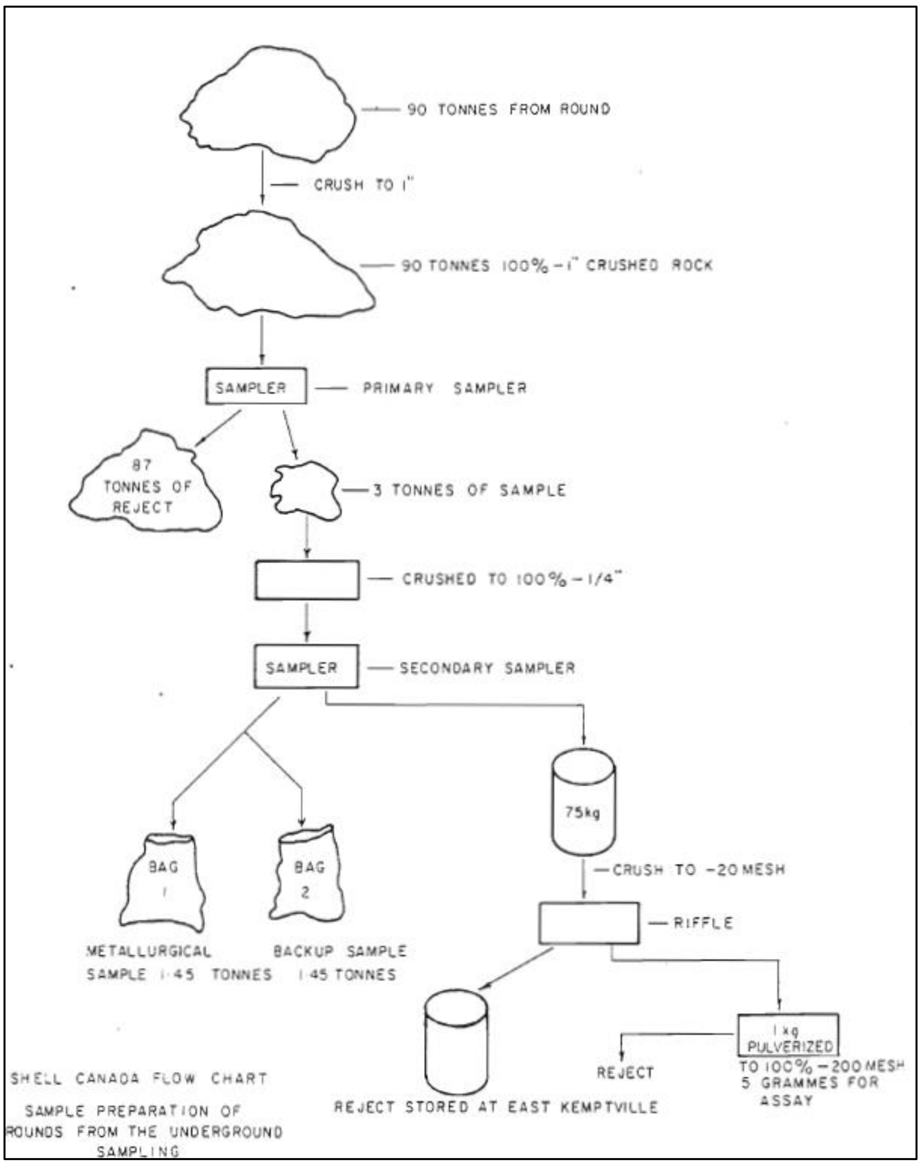
The pressed pellet method used for XRF assays is subject to matrix effects and potential significant variances in accuracy and precision. A high number of repeat assays are typically required for quality control/quality assurance purposes. It is noted that 10% of XRF samples were subject to a repeat assay. This is considered good practice for pressed pellet XRF analyses. The use of a single external standard may not be sufficient to cover the range of expected assay values. The use of an internal Sb standard in the sample is subject to laboratory preparation error.

Overall, review of the sample preparation and analytical procedures employed by Shell and RAL are regarded as consistent with best practice at the time. The results obtained are regarded as reliable within the limits of detection and accuracy available at the time.

12.8 UNDERGROUND BULK SAMPLING

The Shell underground bulk sample program was designed to provide a bulk sample for metallurgical testing and to confirm the grades indicated by diamond drilling. The bulk sample recovered about 31,600 t of material from 298 rounds. An on-site sampler was used to process the bulk sample to provide a metallurgical sample and a sample for assay. The sampling process was reportedly designed such that the total standard deviation in tin content of a round was less than 5%. The bulk sample processing scheme is illustrated in Figure 12.13. The sample processing procedures employed are considered to be good practice at the time and consistent with current practice.

Figure 12.13
Bulk Sample Processing Scheme – Shell



A split of the final processed sample from each round was sent to XRAL (XRF analysis) and Lakefield Research Limited (AA analysis). (Lakefield Research is the predecessor company to SGS Lakefield Canada). The analytical data showed excellent correlation, as illustrated in Table 12.13. The original data has not been verified, but there is no reason to believe the data are not accurate.

Table 12.13
Statistical Summary of Round Robin Data – Shell Bulk Sampling Program

Statistic	% Sn		
	XRAL	Lakefield	Average
Mean	0.199	0.203	0.201
SD	0.126	0.124	0.125
Variance	0.016	0.015	0.0155
Standard Error	0.007	0.007	0.007
Number of Rounds	289	289	289

Source: RAL, 1983.

12.9 SHELL AND RAL CHECK ASSAY PROGRAMS

Both Shell and RAL undertook extensive programs of check assays and inter-laboratory comparisons as part of their exploration and due diligence programs.

12.9.1 Pulp Check Assays (Shell)

Shell sent 114 sample pulps to XRAL and Bondar Clegg to compare with the original Atlantic Analytical result. The analytical methods used by XRAL and Bondar Clegg were as discussed above.

12.9.2 BQ Check Assays (RAL)

RAL sent 160 samples of Shell's remaining split BQ core to XRAL as a check of possible bias introduced by core splitting and/or sample preparation. There was one significant outlier in the assays results.

RAL checked the repeatability of the XRAL results by resubmitting 16 BQ sample pulps under new sample numbers.

RAL submitted 10 HQ sample pulps (-200 mesh) to XRAL, the Wheal Jane tin mine laboratory in Cornwall, United Kingdom, and Lakefield Research.

RAL evaluated the effectiveness of its HQ drill core sample preparation procedures by assaying the coarse (¼-in) rejects. Seven assays by XRAL and five assays by Lakefield Research were completed.

12.10 VERIFICATION OF HISTORIC SAMPLING AND ASSAY DATA

Data verification of the historic sampling and assay data consisted of checking the reported assay values contained in the QA/QC appendix of the RAL feasibility study against the current assay data base and the available drill logs. The current assay database is a compilation undertaken by Avalon of all available assay certificates, drill logs and survey data.

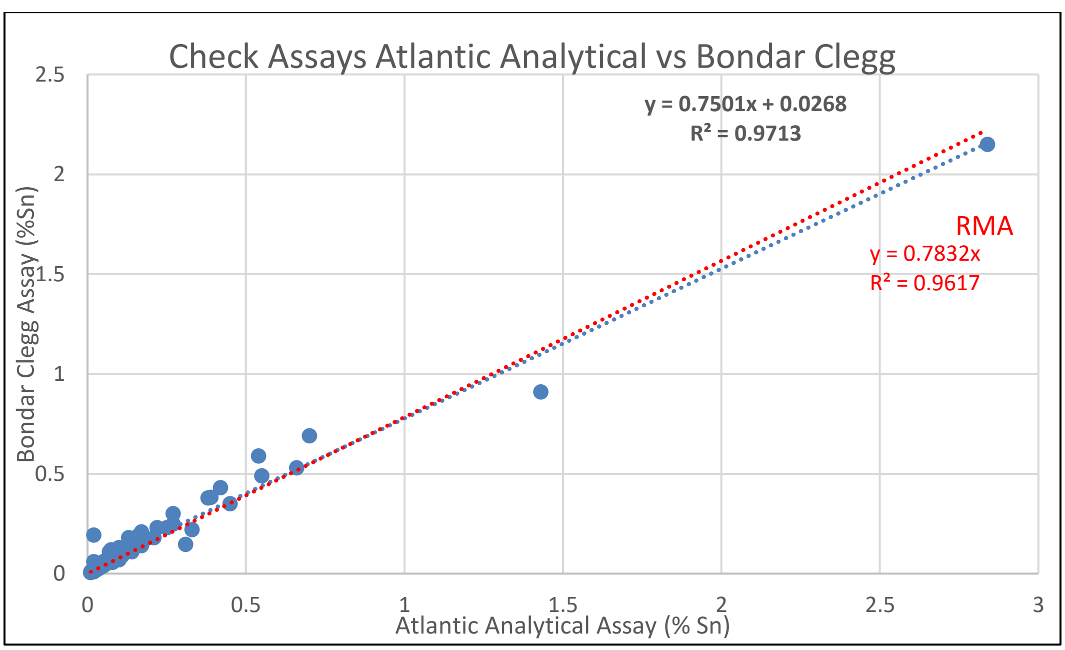
The results of the data verification of historic QA/QC procedures are summarized below.

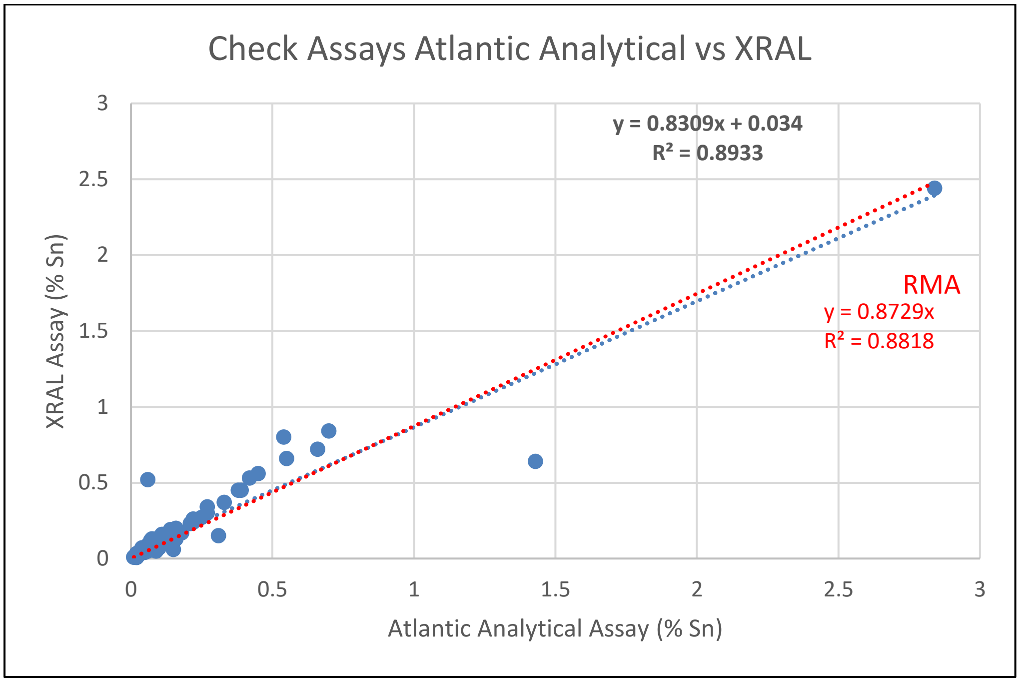
12.10.1 Pulp Check Assays

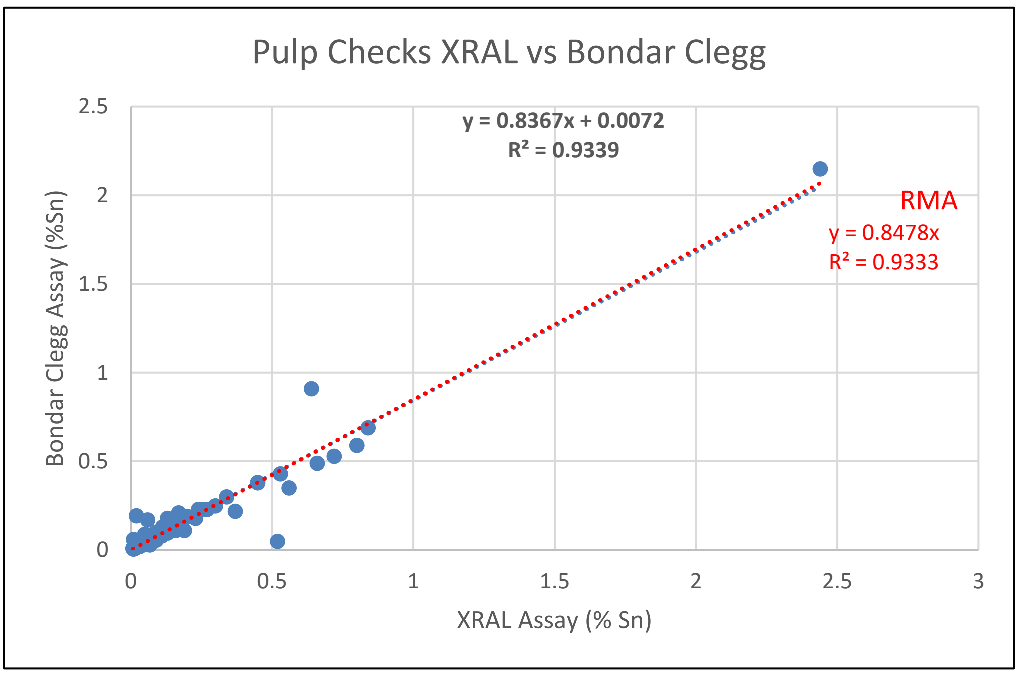
One hundred and four of the 114 original pulp check assays could be verified against the current assay database, original drill logs and assay certificates. Five samples reported assay values from the database different from the values in the data provided in the RAL feasibility study. These results were adjusted to the values for the samples in the database. A number of other samples reported values differing from the database values due to rounding. All other values matched the database values.

Regression analysis (Figure 12.14) shows a high degree of correlation between the assay results, with excellent agreement between the Atlantic Analytical results versus Bondar Clegg results, and XRAL versus Bondar Clegg results. The agreement between the Atlantic Analytical results and the XRAL results is somewhat less but still within acceptable limits. There would appear to be a slight negative bias for the Atlantic Analytical results versus either XRAL or Bondar Clegg based on the Reduced to Major Axis (RMA) analysis.

Figure 12.14
Pulp Check Assays, Shell Minerals Analyses



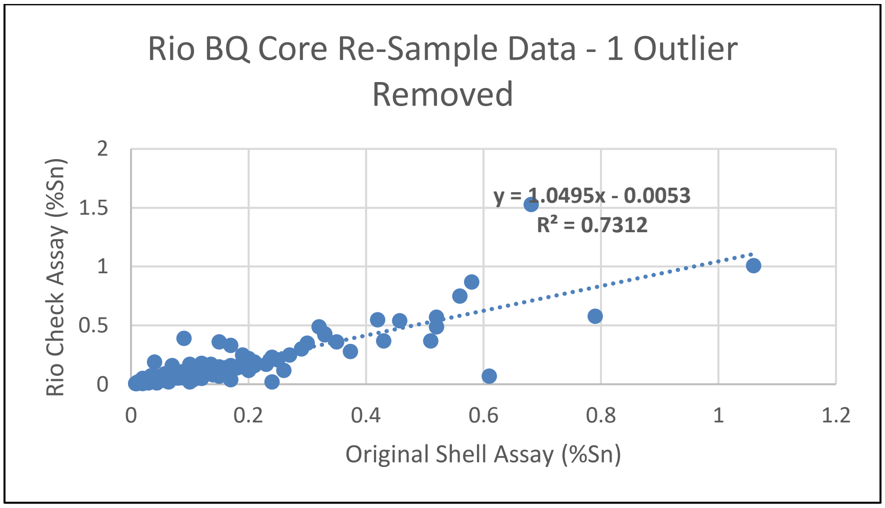




12.10.2 RAL BQ Check Assays

The historic data comprising 160 assays was checked against the current database and one sample assay revised. Minor changes were made to several other historic assays to adjust for rounding. There is one noticeable outlier in the sample data and this was removed from the analysis. Regression analysis of the data (Figure 12.15) shows a moderate correlation (R2 = 0.7312) despite a very small (0.9%) difference in the means. The standard deviation of the sample sets is relatively high, indicating a high degree of scatter in the data, especially with higher assay values. Such a result might be expected given the high nugget texture of the higher-grade samples, especially over short (3-m) sample intervals.

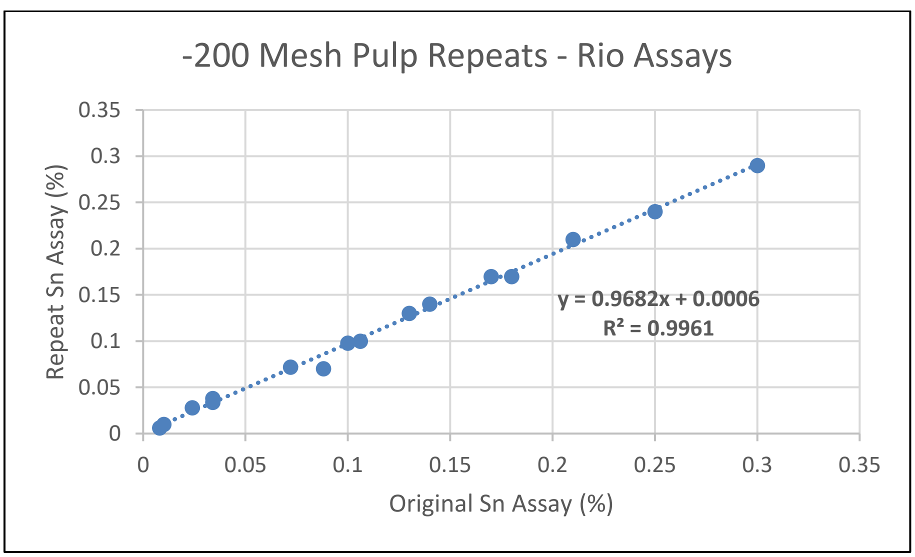
Figure 12.15
RAL BQ Core Check Analyses



12.10.3 RAL BQ Core Pulp Re-Sample

RAL submitted 16 -200 mesh pulps to XRAL for re-assay under new sample numbers. The results of the re-assay indicated excellent agreement (Figure 12.16) and good repeatability.

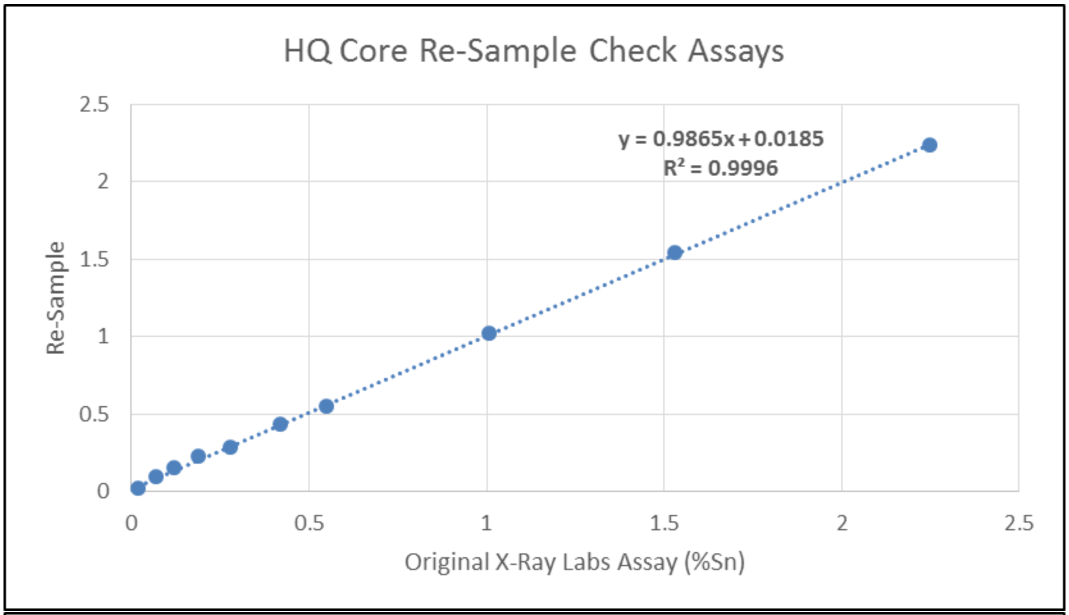
Figure 12.16
BQ Core Pulp Re-sampling, Rio Algom

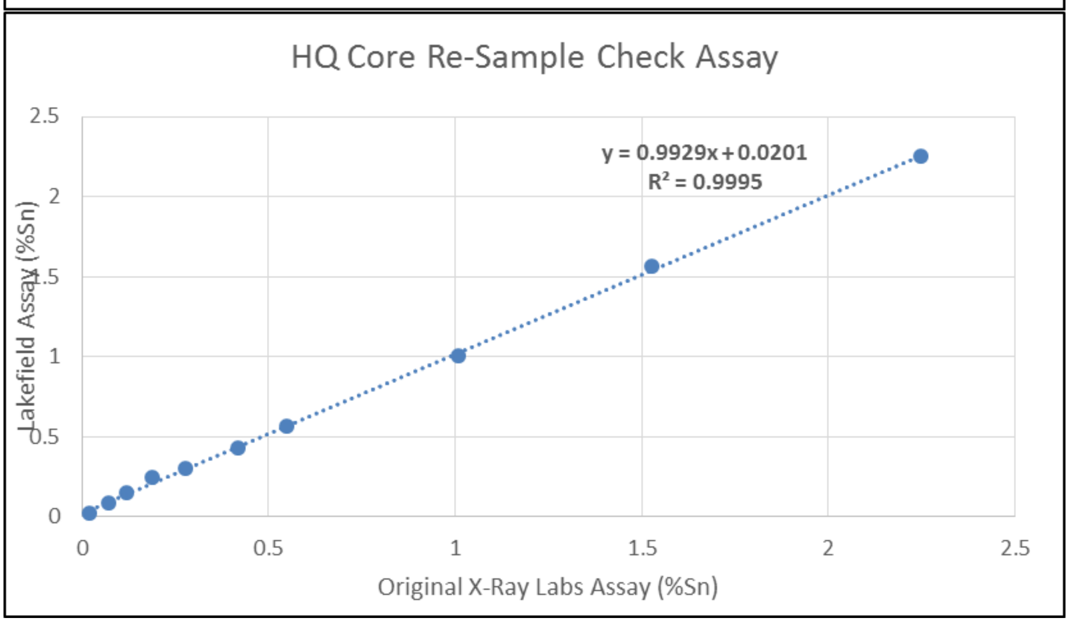


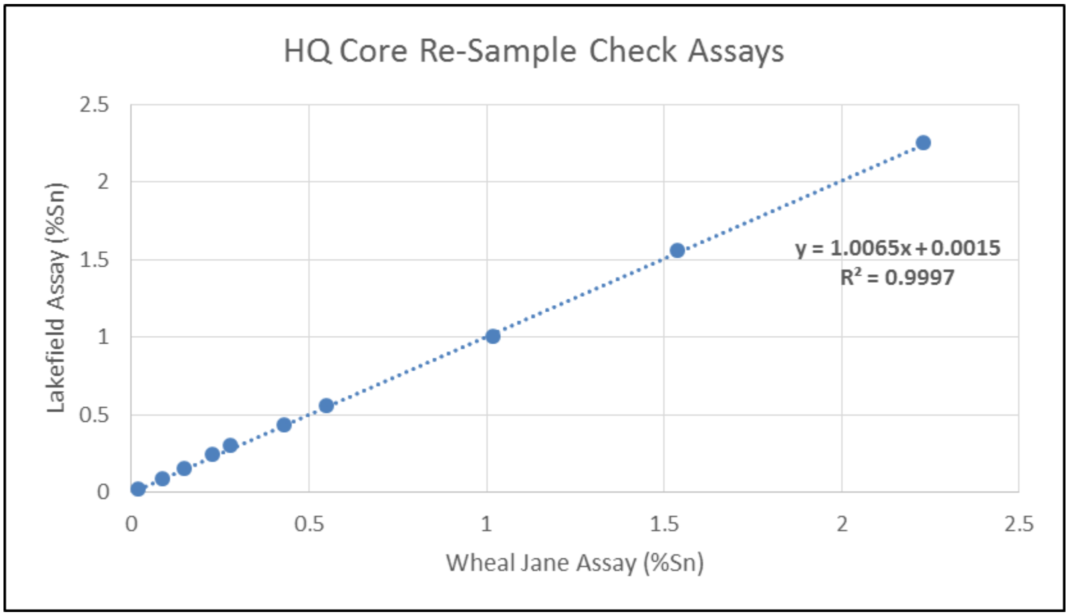
12.10.4 RAL HQ Core Check Samples

RAL checked HQ sample preparation by sending 10 HQ sample pulps (-200 mesh) to each of XRAL, the Wheal Jane mine laboratory and Lakefield Research. The results of the assays (Figure 12.17) show excellent agreement, indicating no bias in final sample preparation.

Figure 12.17
RAL HQ Core Check Samples



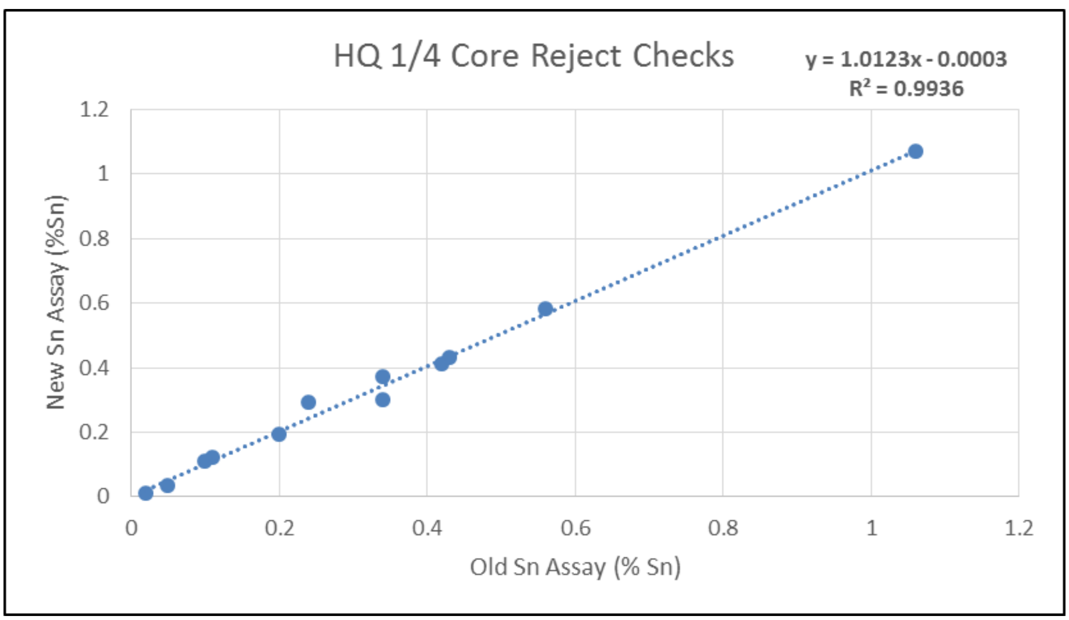




12.10.5 RAL HQ Core ¼-in Rejects

Initial sample preparation quality control issues were evaluated by running check assays on 12 samples of ¼-in HQ reject core. Seven samples were analysed at XRAL and five at Lakefield Research. The results showed excellent agreement (See Figure 12.18).

Figure 12.18
RAL HQ Core ¼-in Reject Check Samples



12.11 Conclusions – Shell and RAL Sampling Methods, Sample Preparation and Assays – Historic Exploration Drilling

Review of the available data indicates that the sampling methods, sample preparation and assaying methods employed by Shell and RAL were consistent with good practice at the time. In comparison with current practice, the number of duplicates and check samples is relatively low, as are the number of blanks and certified reference standards. Despite these limitations, the methodologies employed by Shell and RAL do provide reasonable assurance that no systematic bias was present in either the sample preparation or assaying.

12.12 Avalon Historic Drill Core Re-Sampling Program

Avalon undertook a program of drill core resampling in 2007 and 2008. Selected historic drill core stored at the NSDNR core storage facility in Stellarton, Nova Scotia was either whole core sampled, ½-core or ¼-core sampled, depending on the availability of material and recommendations from NSDNR staff (in some cases, core had previously been reduced to quarter samples and the remaining core was sampled). The resampling program was designed to cover the area of the Baby Pit and various unmined mineralized zones in the Main Pit (2008). Table 12.14 summarizes the selected core and the objectives of resampling the selected intervals.

Figure 12.19 and Figure 12.20 illustrate the locations of the re-sampled drill holes. In total, 526.29 m of drill core was re-sampled, representing 404 samples, including seven field duplicates.

The 2007 re-sample program included a small test program of 13 samples to evaluate analytical methods for indium. This work was undertaken at SGS in Don Mills (SGS Don Mills), Ontario, with check assays at Actlabs in Ancaster, Ontario.

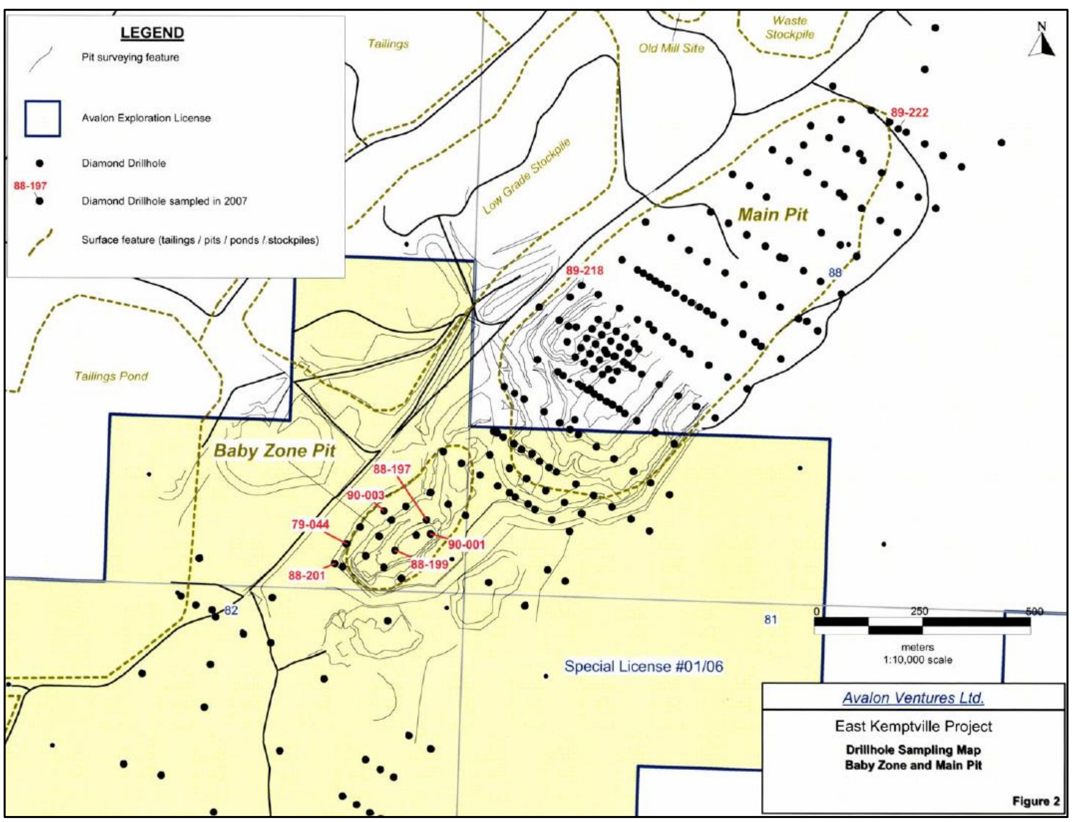
Table 12.14
Avalon Drill Core Re-Sample Program Summary

Hole ID	From (m)	To (m)	Interval (m)	% Sampled	% Left	No. of Samples	Objective
79-044	110.00	155.00	45.00	¼ BQ	¼ BQ	30	Consecutively sample wide, lower grade Sn- and Zn-rich interval in Baby Zone for In. Confirm RAL analysis.
88-197	18.50	54.50	17.50 (49%)	½ BQ	None	5	Sample portions of remaining core from near surface, higher grade Sn and Zn intervals in Baby Zone for In. Confirm RAL analysis. Sections for these holes previously sampled so whole core sampling allowed by NSDNR.
88-199	6.10	60.00	26.90 (55%)	½ BQ	None	11	
88-201	74.10	86.10	12.00	¼ BQ	¼ BQ	6	Consecutively sample higher-grade Zn-rich interval adjacent to Baby Zone in shear zone/quartz veins within sediments for Indium. Confirm RAL analysis.
90-001	149.97	197.50	47.53	½ HQ	¼ HQ	33	Consecutively sample wide, lower grade Sn and Cu-rich intervals below Baby Zone for Indium. Confirm RAL analysis. Initial anomalous Avalon In analysis from short Zn-rich grab samples from this interval. Section represents either deeper footwall to intrusions or possibly a structural dislocation of the Baby Zone or other high-level intrusive.
90-003	55.00	176.00	121.00	¼ BQ	¼ BQ	83	Consecutively sample wide, low to higher grade Sn-Zn-Cu

Hole ID	From (m)	To (m)	Interval (m)	% Sampled	% Left	No. of Samples	Objective
							rich sections in thickest portion of Baby Zone for In. Confirm RAL analysis.
89-218 (Main)	82.00	88.00	6.00	¼ BQ	¼ BQ	2	Sample Zn-rich small sections of core from Main Pit for In.
89-222 (Main)	69.00	72.00	3.00	¼ BQ	¼ BQ	1	Sample Zn-rich small section of core from Main Pit for In.
80-093	55.50	99.10	40.60 (100%)	¼ BQ	¼ BQ	27	Consecutively sample unmined, wide, medium to higher grade Sn intervals in Southwest Extension Zone and below Main Pit likely to be mined. Intercepts selected based on remaining drill core at NSDNR Core Library and modelled pit surface based on best survey maps available at the time.
89-204	4.20	105.00	100.80 (100%)	½ BQ	¼ BQ	67	
89-216	1.00	65.00	64.00 (100%)	½ BQ	¼ BQ	43	
89-225	84.00	175.26	90.26 (98.9%)	¼ BQ	¼ BQ	61	
89.229	34.00	85.00	51.00 (100%)	¼ BQ	¼ BQ	35	
Total sampling Baby Zone & Main Pit – 2007			278.93			171	Incl. 5 field duplicates plus standards and blanks.
Total sampling Southwest Extension and Main Pit – 2008			247.36			233	Incl. 2 field duplicates plus standards and blanks.
Total Sampling 2007 and 2008			**526.29**			**404**	**Incl. 7 field duplicates plus standards and blanks**

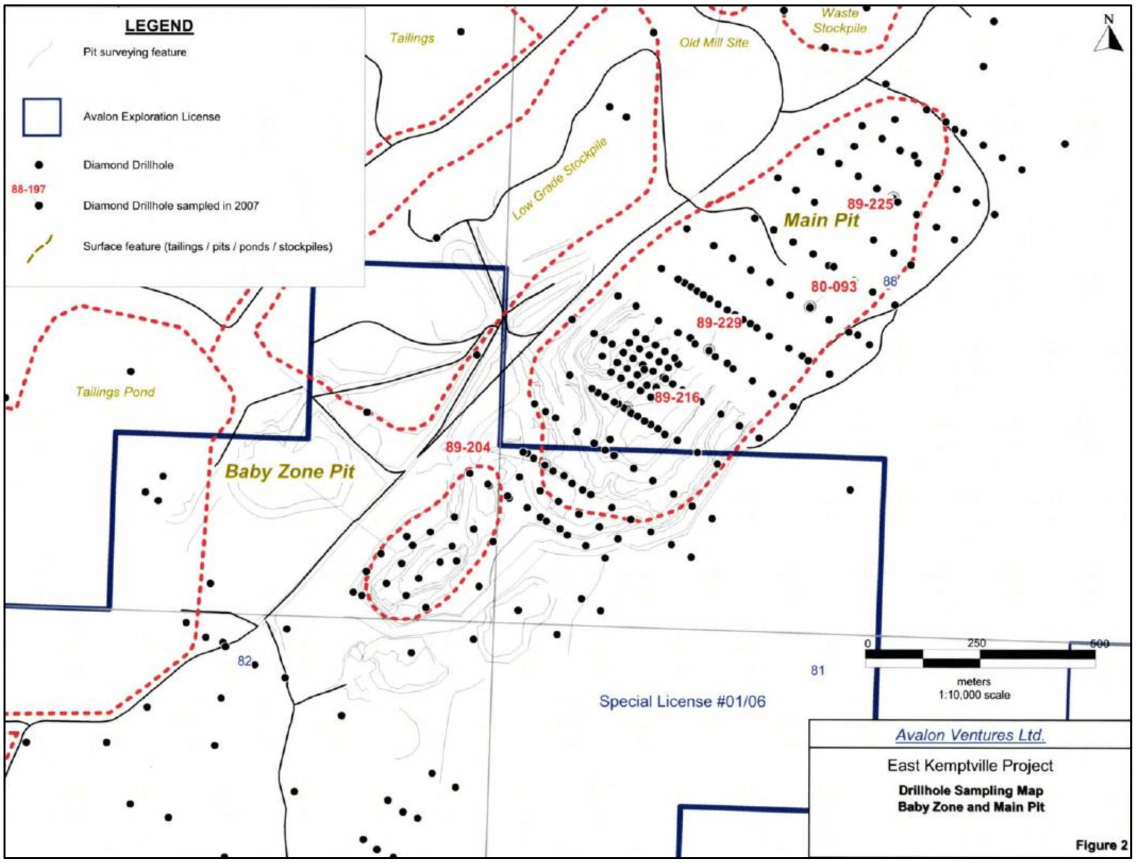
Source: Avalon Assessment Reports, 2007 and 2008.

Figure 12.19
Drill Core Re-Sample Program 2007



Source: Avalon, 2007.

Figure 12.20
Drill Core Re-Sample Program 2008



Source: Avalon, 2008.

12.12.1 Sample Preparation and Analysis: 2007 and 2008 Re-Sample Program

Initial sample preparation (13 samples) for the 2007 program consisted of crushing to 75% passing 2 mm, riffle splitting a 250 g sample, pulverizing to -200 mesh, followed by splitting of an analytical sample and multi-element analysis by sodium peroxide fusion followed by ICP-OES and ICP-MS (ICM90A method). Analysis at Actlabs was by XRF (for Sn), multi-element assays by fusion/ICP and fusion-MS. Zinc assays were by ICP-OES and indium assays by INAA.

The main analytical work comprised 159 samples which were crushed and pulverized at the SGS facility at Lakefield, and tin assays performed by XRF. Representative sample pulp splits were analysed at the SGS facility at Don Mills using the multi-element ICM90A method.

Sample preparation and assaying for the 2008 re-sample program consisted of crushing, splitting, pulverizing and assaying for tin by XRF at SGS Lakefield, with representative sample splits assayed at SGS Don Mills using the ICPM90A method.

12.12.2 QA/QC Re-Sample Program

QA/QC work for the 2007 re-sample program involved development of two independent indium standards by blending reject samples from the initial 13 sample test program. These standards were subsequently inserted into the main sample batch. Two commercial indium standards (origin unknown) were used to monitor laboratory performance in the initial test program.

The main 2007 re-sample program incorporated five blanks, 12 standards and five field duplicates. The standards comprised commercially produced material and standards prepared by blending homogenized material form the testwork. Blanks were prepared from unmineralized drill core material. No round robin assay program was undertaken on the blanks or non-commercial standards.

The 2008 re-sample program incorporated eight blanks, nine standards (commercial and internally produced) and two field duplicates.

Available data on the blanks and standards produced by Avalon are discussed in Section 11.1.1 and shown in Table 12.4.

The use of uncertified blanks and standards in the re-sample program is considered to be acceptable given the purpose of the program.

12.12.3 Avalon Drill Core Re-Sampling Program Results

Table 12.16 summarizes the results of the 2007 and 2008 drill core re-sampling programs. The original data has been verified by checking the assay certificates and the current Avalon assay database for the East Kemptville Project. The data indicates a high degree of correlation between the RAL assay results and the Avalon re-sample assay results. The results confirm the historic check assay and due diligence results obtained by RAL in its review of the Shell results. Based on the results, the Shell and RAL drill core assays are considered as acceptable for resource estimation purposes.

Table 12.15
Summary Results – Comparison of RAL Assays with Avalon Re-Sample Assays

Hole ID	Avalon	RAL	Avalon	RAL	Avalon	RAL
	Sn (ppm)		Zn (ppm)		Cu (ppm)	
2007 Program						
79-044 (15 intervals)	672	756	2,351	2,357	162	100
88-197 (5 intervals)	1,462	1,674	2,321	1,880	516	520
88-199 (9 intervals)	2,193	2,349	5,084	4,522	401	344
88-201 (3 intervals)	693	500	3,200	5,200	419	533
90-001 (15 intervals)	1,184	1,100	2,982	3,200	586	673
90-003 (39 intervals)	1,600	1,599	2,695	3,008	537	664
Average of 86 intervals	1,770	2,572	2,935	3,056	460	521
Average of 86 intervals excl. 1 outlier	1,396	1,416				

Hole ID	Avalon	RAL	Avalon	RAL	Avalon	RAL
	Sn (ppm)		Zn (ppm)		Cu (ppm)	
2008 Program						
80-093 (14 intervals)	2,176	2,370	2,428	2,416	995	797
89-204 (34 intervals)	2,831	3,112	1,491	1,783	1,349	1,402
89-216 (22 intervals)	1,712	1,996	612	670	1,083	1,180
89-225 928 intervals)	1,812	1,425	1,122	1,089	474	546
89-229 (17 intervals)	3,155	4,029	3,129	3,141	881	988
Average of 115 intervals	2,356	2,533	1,587	1,679	977	1,020

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Avalon has conducted a number of testwork programs on the East Kemptville deposit in recent years. Work began with SGS UK in Cornwall, UK to develop a more comprehensive flowsheet to produce tin, copper, and zinc/indium concentrates using mineralized samples from the Baby Zone deposit. This set the baseline for a subsequent test program campaign in 2016 at Met-Solve who investigated recovering tin (only) from the existing low-grade ore stockpile. Table 13.1 below lists all the flotation/concentrator testwork reports issued as part of these two programs.

Table 13.1
List of Testwork Relevant to Stockpile Flowsheet Development

Date	Author	Title	Remarks
January, 2016	SGS UK	East Kemptville Phased Process Development Test Work Phase 1 and Phase II Interim Report	Testwork by SGS UK to investigate recovering tin, copper, and zinc from the Baby Zone
October, 2016	Met-Solve	Avalon East Kemptville Gravity Testwork Report	Testwork by Met-Solve on low grade tin stockpile material from East Kemptville
February, 2017	Met-Solve	Avalon East Kemptville Gravity Testwork Supplementary Report	Supplementary report on Oct 2016 testwork with additional results from Locked Cycle Tests (LCT)

13.2 SGS UK 2015 TESTWORK

Avalon engaged SGS UK in Cornwall to undertake testwork on a sample of mineralization collected from the Baby Zone. The testwork program was divided into two phases. The first phase looked at using heavy liquid separation and gravity pre-concentration to remove barren waste prior to down-stream processing. The second phase developed a process using sulphide flotation for copper/zinc recovery followed by gravity concentration and flotation for tin recovery. The findings in this work provided the basis of investigation for later low-grade stockpile testwork programs by Met-Solve.

The narrative below summarizes those results relevant to tin recovery and have been used in the development of the proposed flowsheet.

13.2.1 Metallurgical Samples

A total of 1,140 kg of material comprising 394 split drill core from the Baby Zone were sent to SGS UK. A composite was prepared using 94 samples originating from the 2014 drilling campaign. These samples were selected by Avalon from two drill holes, EKAV-14-002 and 003. The testwork composite weighed approximately 290 kg and, although relatively high grade, is considered representative of the typical mineralization contained within the Baby Zone mineral resource. A head sample analysis of the composite is presented in Table 13.2.

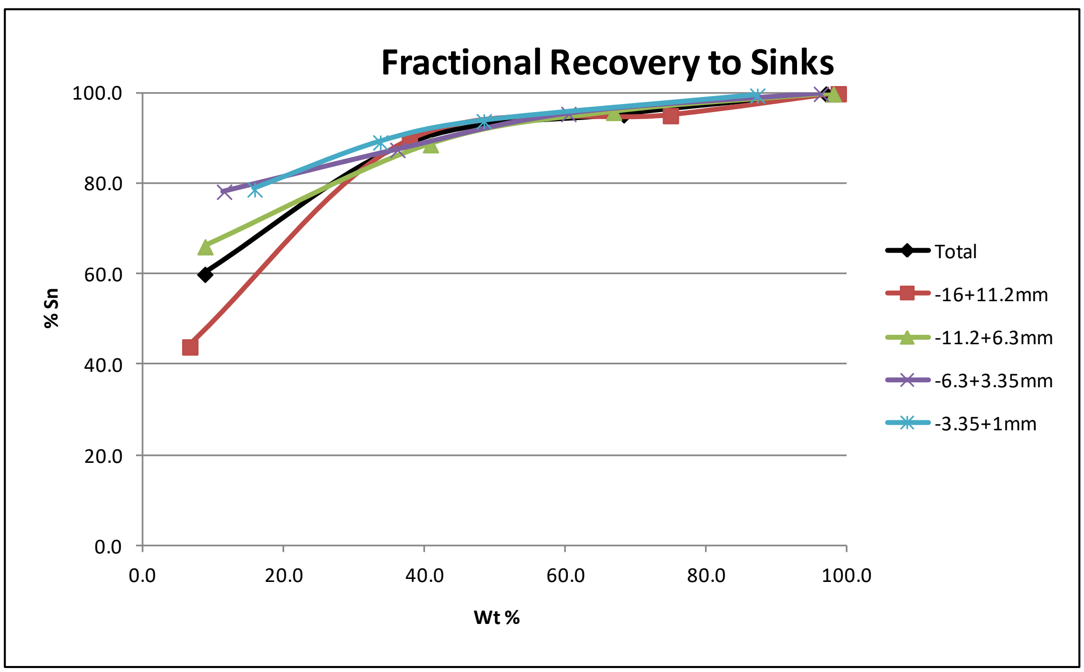
Table 13.2
SGS UK Metallurgical Composite Head Assay Results

Cu (%)	Zn (%)	Fe (%)	As (%)	S (%)	Sn (%)	In (%)	Au (ppm)	Ag (ppm)
0.090	0.44	2.35	0.020	1.67	0.34	<0.001	0.08	2.5

13.2.2 Heavy Liquid Testing

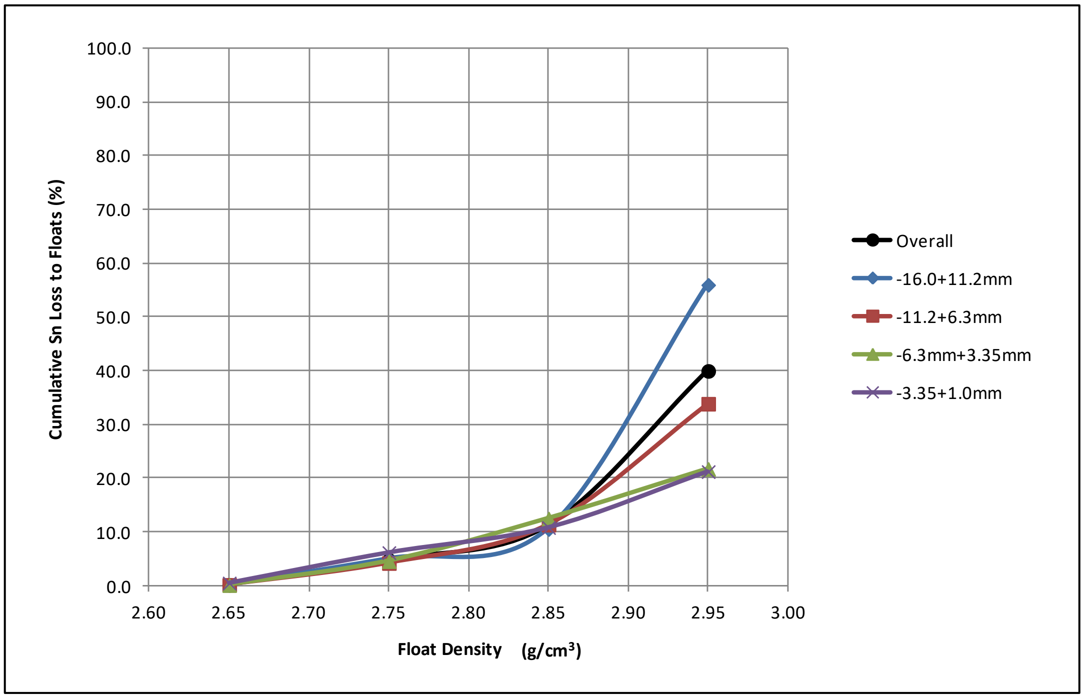
Four size fractions (-16.0 mm + 11.2 mm, -11.2 mm + 6.3 mm, -6.3 mm + 3.35 mm and -3.35 mm +1 mm) were tested using heavy liquid sink/float separation at 2.65 g/cm^3, 2.75 g/cm^3, 2.85 g/cm^3 and 2.95 g/cm^3. The products from each of the four test densities were collected and assayed for copper, zinc, and tin. Copper and zinc recovery behaved similarly, but quite different from tin. Results for the tin mass pull recovery can be seen below in Figure 13.1. This indicates that ~95% of the tin can be recovered in ~50% of the mass and similarly ~60% tin can be recovered in <10% mass.

Figure 13.1
Tin Mass Pull Recovery



Minimal tin losses occurred until the density was over 2.85 g/cm^3 at all four size fractions (see Figure 13.2). Based on these results, Dense Media Separation (DMS) may be a pre-concentration option to remove waste rock in the crusher circuit, increase plant feed grade and reduce the size of the processing plant. Alternatively, it may be a means for increasing tin production without increasing the size of the processing/gravity plant, and for also upgrading of the very low-grade stockpiles currently not being considered for treatment in this project scenario.

Figure 13.2
Heavy Liquids – Tin Loss to Floats



13.2.3 Bond Rod and Ball Mill Grindability Testwork

Both the Bond rod and ball mill index and an abrasion index tests were completed. Results of these comminution tests are summarized in Table 13.3 below.

Table 13.3
Grinding Test Results

Test Parameter	Product Size (P_{80} - µm)	Results
Bond rod mill index	806	16.2 kWh/t
Bond ball mill index	85	14.5 kWh/t
Abrasion Index		0.584 g

Comparing the grinding index test results with the extensive SGS UK Database it can be concluded that the mineralization from the Baby Zone is slightly harder than average. The abrasion index suggests that the material is relatively abrasive, probably due to the siliceous nature of the mineralization.

13.2.4 Test Results – Gravity Separation Using Mozley Concentrators

A number of gravity separation tests were conducted on bulk flotation tailings solids by SGS UK to investigate the possibility of using Mozley Concentrators to recover the tin.

The initial set of tests were done on 200-micron tailings solids after bulk flotation, and it was found that laboratory gravity separation was able to achieve a tin concentration upgrade ratio of at least 5:1 compared to the head grade. Results are seen in Table 13.4 below.

Table 13.4
200-Micron Grind Flotation Tailings Gravity Separation Results

Test F5		Assay (%)				Recovery (%)			
Product	Mass (%)	Cu	Zn	S	Sn	Cu	Zn	S	Sn
Bulk float con	4.53	1.900	10.52	34.99	0.25	92.9	87.1	88.5	4.1
Float tailings	95.47	0.007	0.074	0.19	0.28	7.1	12.9	11.5	95.9
Grav. Ro. Con.	5.47	0.009	0.130	0.43	1.40	0.5	1.3	1.5	27.2
Grav. Ro. Mids.	39.98	0.009	0.084	0.24	0.36	3.9	6.1	6.0	51.0
Grav. Ro Tails	50.01	0.005	0.060	0.13	0.10	2.7	5.5	4.1	17.7
Float feed	100.00	0.090	0.550	1.60	0.28	100.0	100.0	100.0	100.0

The material, with a Sn head grade of 0.28% Sn was concentrated to 1.4% Sn in the concentrate with 27.2% Sn recovery. Most of the remaining tin was locked in the middlings where the tin recoveries were 51%, which suggested that more milling may be necessary to further liberate the tin minerals for concentration. Overall recovery to a combined concentrate plus middlings product was 87.2%.

With these results, in the next set of testwork, SGS UK milled the bulk flotation tailings to 80% passing 50 microns to see if a finer grind would improve tin recovery in the concentrates. Results for the 50-micron gravity separation testwork can be seen in Table 13.5 below.

Table 13.5
50-micron Grind – Gravity Separation Results

Test F6		Assay (%)				Recovery (%)			
Product	Mass (%)	Cu	Zn	S	Sn	Cu	Zn	S	Sn
Bulk Float Con.[1]	7.9	1.22	15.40	17.77	0.16	89.9	87.0	82.7	5.0
Float Tailings[1]	92.10	0.012	0.199	0.32	0.38	10.1	13.0	17.3	95.0
Grav. Ro. Con.	3.00	0.012	0.016	3.32	6.57	0.3	0.0	5.8	53.6
Grav. Ro. Mids.	6.50	0.009	0.015	1.75	0.53	0.5	0.1	6.7	9.5
Grav. Ro Tails	82.60	0.012	0.220	0.10	0.14	9.3	12.9	4.8	31.9
Float Feed	100.00	0.107	1.400	1.70	0.36	100.0	100.0	100.0	100.0

[1]Back calculated from gravity results and float feed calculated head assays.

With the finer grind, tin recovery increased from 25% to 54% in the gravity concentrates and grade improved to 6.57% Sn. Recovery of tin to the middlings also decreased as expected to

9.5%. However, loss of tin to the tailings increased to 32% from 18%, signaling that 50 microns may be too fine of a grind and more tin was being lost to the tailings stream as fines.

13.2.5 Test Results – Gravity Separation Using Shaking Tables

Following the encouraging results with the finer grind gravity test, SGS UK repeated the testwork using more conventional shaking tables (see Figure 13.3). Tests were undertaken at 200, 100 and 50 microns to assess the viability of using shaking tables to concentrate tin from bulk flotation tailings.

Figure 13.3
Picture of a 200 Micron Gravity Separation Test

Initial results with the 200-micron material showed improved results to those using Mozley Concentrators. Approximately 72% of the tin was recovered in the rougher concentrate with a grade of 6.6% Sn. Results are summarized in Table 13.6.

Table 13.6
200 Micron Primary Gravity Separation Shaking Table Test Results

-200 Micron		Assay (%)				Recovery (%)			
Product	**Mass (%)**	**Cu**	**Zn**	**S**	**Sn**	**Cu**	**Zn**	**S**	**Sn**
Grav. Ro. Con.	4.40	0.014	0.280	2.49	**6.61**	5.3	10.9	30.2	**71.6**
Grav. Ro Tails	95.60	0.012	0.105	0.266	0.121	94.7	89.1	69.8	28.4
Test Feed	100.00	0.012	0.113	0.364	0.408	100.0	100.0	100.0	100.0

It should be noted that a middlings stream was not produced as the objective of the testwork was to produce a higher recovery rougher concentrate suitable for further cleaning into a saleable product, leaving tailings for further regrinding and increased tin release. Physically, it is also easy to see the darker tin stream, enabling better control for operators during operations.

Table 13.7 presents the results of gravity separation of reground -200-micron test tailings. A scavenger concentrate containing 1.6% Sn was produced with a 34% recovery.

Table 13.7
100 Micron Scavenger Gravity Test Results After Regrinding 200 Micron Rougher Test Tailings

-100 Micron		Assay (%)				Recovery (%)			
Product	**Mass (%)**	**Cu**	**Zn**	**S**	**Sn**	**Cu**	**Zn**	**S**	**Sn**
Grav. Sc. Con.	2.60	0.019	0.068	1.74	**1.59**	4.3	1.7	17.0	**34.1**
Grav. Sc Tails	97.40	0.011	0.106	0.227	0.082	95.7	98.3	83.0	65.9
Test Feed[1]	100.00	0.012	0.105	0.266	0.121	100.0	100.0	100.0	100.0

[1]Test feed is the reground tailings from the 200-micron rougher test.

This additional gravity test increases overall tin recovery to 81.3%.

SGS then took the rougher tailings from the 100-micron gravity testwork and further milled them to 50 microns and conducted additional gravity separation on the material. Results can be seen in Table 13.8 below.

Table 13.8
50 Micron Secondary Scavenger Gravity Testwork Results After Regrinding 100 Micron Scavenger Test Tailings

-50 Micron		Assay (%)				Recovery (%)			
Product	**Mass (%)**	**Cu**	**Zn**	**S**	**Sn**	**Cu**	**Zn**	**S**	**Sn**
Grav. Sc. Con.	4.90	0.018	0.330	0.68	0.34	7.7	15.1	14.6	20.2
Grav. Sc Tails	95.10	0.011	0.095	0.203	0.069	92.3	84.9	85.4	79.8
Test Feed[1]	100.00	0.011	0.106	0.227	0.082	100.0	100.0	100.0	100.0

[1]Test feed is the reground tailings from the 100-micron scavenger test.

Given that tin head grades were low after two cycles of gravity separation, only 20.2% of the tin was recovered into a 0.34% Sn concentrate. This translates to an additional 3.8% increase in overall tin recovery, which may not merit the additional milling and gravity separation stage on the 100-micron scavenger tailings.

13.2.6 Test Results – Tin Flotation

Tin flotation tests were carried out on the various gravity tailings using the conditions outlined in Table 13.9 below.

Table 13.9
Tin Flotation Conditions

Sulphide Scavenger	FT12	FT13	Tin Rougher	FT12	FT13
pH	5.5	5.5	pH	5.2	5.2
H_2SO_4 g/t	31	31	H_2SO_4 g/t	156	156
$CuSO_4$ g/t	250	250	Sodium Silicate g/t	1000	1000
SIPX g/t	100	100	Aluminum Sulphate g/t	-	-
MIBC g/t	20	20	Aero 845 g/t	330	330
Condition Time	5/2 min	5/2 min	Condition Time	5/5 min	5/5 min
Flotation Time	2	2	Flotation Time	15	15

The feed to test FT12 was scavenged 50-micron tailings, and FT13 used 100-micron gravity tailings milled to 50 microns to compare gravity/flotation and flotation only.

The flotation feed was subjected to desliming at approximately 8 microns and a sulphide scavenger flotation prior to tin flotation.

The tin flotation rougher produced a tin stage recovery of 63%, with a maximum grade of 0.38% Sn from a 0.069% Sn head grade. Flotation was essentially complete after 10 min, but since the feed only represented 14.9% of the tin in the original feed, the overall tin recovery was 9.4%. Some tin was able to be extracted from the 50-micron tailings, but the grades and recovery were low. See Table 13.10.

Flotation was also conducted on the 50-micron material produced without a gravity scavenger circuit in test FT13. Results are shown in Table 13.11.

Flotation without a prior gravity scavenger stage did not produce significantly better results than FT12, which included gravity scavenging. Tin recovery was high at 67% after 5 rougher stages, but overall rougher concentrate tin grades were low at 0.14%. The feed only represented 18.5% of the tin in plant feed, so the overall tin recovery was 12.4%.

Table 13.10
50 Micron Gravity Separation Tailings Tin Flotation Results

Test FT12		Assay (%)				Recovery (%)			
Product	**Mass (%)**	**Cu**	**Zn**	**S**	**Sn**	**Cu**	**Zn**	**S**	**Sn**
Slimes	10.9	0.014	0.12	0.20	0.13	13.9	13.8	10.7	20.5
Sulphide Con	0.6	0.690	12.93	10.10	0.27	37.0	80.0	29.2	2.3
Sn Float Ro. Con.	23.4	0.015	0.020	0.37	**0.19**	31.3	4.2	42.1	**63.0**
Sn Flotation Tails	65.1	0.003	0.003	0.06	0.02	17.8	2.1	17.9	14.2
Float Feed	100.0	0.011	0.100	0.20	0.069	100.0	100.0	100.0	100.0

Table 13.11
50 Micron Tin Flotation Without Gravity Separation

Test FT13		Assay (%)				Recovery (%)			
Product	**Mass (%)**	**Cu**	**Zn**	**S**	**Sn**	**Cu**	**Zn**	**S**	**Sn**
Slimes	11.7	0.01	0.12	0.22	0.12	15.4	16.1	13.8	18.4
Sulphide Con	0.7	0.420	9.690	7.18	0.20	37.7	75.8	26.2	1.8
Sn Float Ro. Con.	37.0	0.008	0.010	0.26	**0.14**	40.2	4.6	51.4	**66.6**
Sn Flotation Tails	50.6	0.001	0.010	0.03	0.02	6.7	3.5	8.6	13.2
Float Feed	100.0	0.010	0.090	0.19	0.077	100.0	100.0	100.0	100.0

13.2.7 Gravity Concentrate Cleaning Tests

Rougher gravity concentrates from the 200, 100, and 50-micron shaking table tests were cleaned by using a Mozley Table. Results for these tests are shown in Table 13.12, Table 13.13 and Table 13.14. The corresponding feed samples relate to the gravity rougher and scavenger test results shown in Table 13.6, Table 13.7 and Table 13.8 above.

Table 13.12
200 Micron Grind Cleaner Gravity Separation Results

200 Micron Cleaner Test		Assay (%)				Recovery (%)			
Product	**Mass (%)**	**Cu**	**Zn**	**S**	**Sn**	**Cu**	**Zn**	**S**	**Sn**
Grav. Ro. Con.	38.0	0.013	0.240	2.19	**14.22**	34.8	34.3	32.0	**84.7**
Grav. Ro. Mids.	26.0	0.012	0.270	3.65	2.60	22.0	26.4	36.5	10.6
Grav. Ro Tails	36.0	0.017	0.290	2.28	0.84	43.2	39.3	31.6	4.7
Test Feed (Ro. Con.)	100.0	0.014	0.270	2.60	6.38	100.0	100.0	100.0	100.0

Table 13.13
100 Micron Grind Cleaner Gravity Separation Results

100 Micron Cleaner Test		Assay (%)				Recovery (%)			
Product	**Mass (%)**	**Cu**	**Zn**	**S**	**Sn**	**Cu**	**Zn**	**S**	**Sn**
Grav. Ro. Con.	16.7	0.037	0.700	1.89	**8.72**	30.9	22.2	17.7	**76.5**
Grav. Ro. Mids.	35.6	0.020	0.560	1.94	0.73	35.7	37.9	39.0	13.7
Grav. Ro Tails	47.7	0.014	0.440	1.61	0.39	33.4	39.9	43.3	9.8
Test Feed (Ro. Con.)	100.0	0.020	0.530	1.77	1.90	100.0	100.0	100.0	100.0

Table 13.14
50 Micron Grind Cleaner Gravity Separation Results

50 Micron Cleaner Test		Assay (%)				Recovery (%)			
Product	**Mass (%)**	**Cu**	**Zn**	**S**	**Sn**	**Cu**	**Zn**	**S**	**Sn**
Grav. Ro. Con.	4.4	0.057	0.069	1.35	**3.44**	11.9	1.4	9.0	**55.5**
Grav. Ro. Mids.	10.2	0.090	0.064	1.11	0.35	43.6	3.0	17.2	13.1
Grav. Ro Tails	85.4	0.011	0.240	0.57	0.10	44.5	95.5	73.8	31.4
Test Feed (Ro. Con.)	100.0	0.021	0.210	0.66	0.27	100.0	100.0	100.0	100.0

For the -200-micron grind gravity cleaner test 84.7% of the tin was stage recovered to concentrate at a grade of 14.2% Sn, from a feed grade of 6.4% Sn, and a concentrate mass pull of 38%. This demonstrated that at a -200-micron grind, the majority of the tin mineralization in the rougher concentrate was free or liberated and concentrates readily. Tin locking appears to be minimal as the middlings only contained 10.6% of the tin.

The 100-micron testwork showed good tin recoveries (76%) to the concentrate with an 8.7% Sn grade. The cleaner feed grade was 1.90% Sn so there was an upgrading ratio of 4.6.

This feed to the 50-micron cleaner test was the secondary scavenger concentrate generated from regrinding and treating the 100-micron rougher tailings, so the mass pull, head and concentrate grades are low. However, 55.5% of the tin was stage recovered to a concentrate grading 3.4% Sn from a head grade of 0.27% Sn and a mass pull of 4.4%. The tin mineralization was essentially free or liberated with little signs of locking.

13.2.8 Gravity Cleaner Concentrate Upgrading

The -200 and -100 gravity cleaner concentrates (see Table 13.12 and Table 13.13) were combined and screened at 75 microns; with the oversize being carefully ground until the sample was passing 75 microns, in order to liberate the remaining sulphides without excessive sliming of cassiterite.

The combined sample was then “dressed” by sulphide flotation and magnetics followed by concentrate re-cleaning. The results are shown in Table 13.15.

Sulphide flotation removed 56.3% of the remaining copper, 94.6% of the remaining zinc, 47.2% of the remaining iron and 68.8% of the remaining sulphur at a mass pull of 5.4%. Tin loss to this scavenger sulphide concentrate was only 1.4%.

The tailings from the sulphide scavenger were then subjected to dry LIMS (Low Intensity Magnetic Separation), and dry HIMS (High Intensity Magnetic Separation) to remove any magnetics prior to further gravity cleaning.

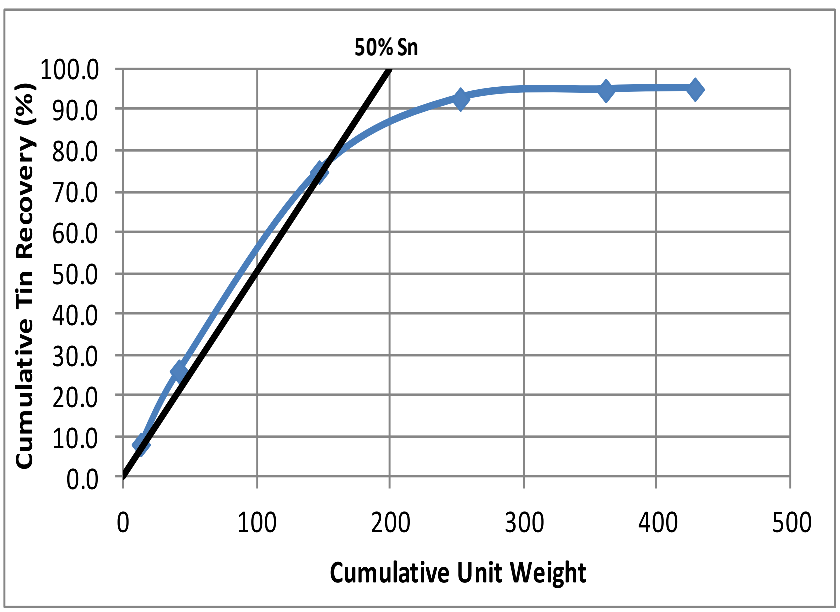
As shown in Table 13.15, 81.1% of the remaining sulphur and 54.2% of the remaining iron were removed by the LIMS, indicating the presence of pyrrhotite, which was not effectively recovered in the sulphide scavenger flotation stage. Zinc and copper were also removed with recoveries of 17.8% and 10.9% respectively, indicating potential fine locking with pyrrhotite. The LIMS magnetic test, which had a concentrate mass pull of 2.3%, removed very little tin, with a stage loss of only 0.3%. The LIMS concentrate assayed 65.5% Fe and 25.2% S, with minimal zinc and copper, demonstrating that a large proportion of the iron was present as magnetite or grinding steel.

The HIMS separation did not add much in the way of impurity removal. Iron and sulphur recoveries were only 10.8% and 4.8% respectively, along with copper at 4.9% and zinc at 2.8%, in a 1% mass pull. However, tin loss was low at 0.3%.

The HIMS concentrate grades were 28.5% Fe and 3.5% S, again indicating that most of the iron removal was of oxides, in this case probably hematite.

Following the sulphide flotation and magnetic removal of impurities, the remaining concentrate then underwent further gravity separation to determine the tin concentrates that could be expected. The test results are shown in Table 13.15 and the tin release curve is shown in Figure 13.4.

Figure 13.4
Final Concentrate Tin Release Curve



The tin release curve showed that the tin is free and easily concentrates with a stage recovery of around 78% to a 50% Sn grade concentrate product.

Table 13.15
Tin Dressing Results

Combined 200 & 100 micron Concentrate Recleaner	Weight	Weight	Assays					Recovery					Cumualtive Assays					Cumulative Recovery				
	g	%	Cu %	Zn %	Fe %	S %	Sn %	Cu %	Zn %	Fe %	S %	Sn %	Cu %	Zn %	Fe %	S %	Sn %	Cu %	Zn %	Fe %	S %	Sn %
Sulphide Concentrate	5.8	5.4	0.25	6.22	43.49	27.63	3.82	56.3	94.6	47.2	68.8	1.4	0.25	6.22	43.49	27.63	3.82	56.3	94.6	47.2	68.8	1.4
Sulphide Tailings	101.9	94.6	0.011	0.02	2.77	0.71	15.47	43.7	5.4	52.8	31.2	98.6										
Feed	107.7	100.0	0.024	0.35	4.96	2.16	14.84	100	100	100	100	100										
LIMS Magnetic Concentrate	2.3	2.3	0.086	0.10	65.48	25.21	2.19	17.8	10.9	54.2	81.1	0.3										
HIMS Magnetic Concentrate	1.0	1.0	0.054	0.06	28.50	3.46	5.05	4.9	2.8	10.3	4.8	0.3	0.076	0.09	54.27	18.62	3.06	22.7	13.8	64.4	86.0	0.7
Non Magnetics	97	96.7	0.009	0.02	1.02	0.10	15.89	77.3	86.2	35.6	14.0	99.3										
Feed	100.3	100.0	0.011	0.02	2.77	0.71	15.47	100.0	100.0	100.0	100.0	100.0										
Recleaner Concentrate 1	1.9	2.0	0.009	0.02	1.08	0.09	67.65	2.0	1.6	2.1	1.7	8.3	0.009	0.02	1.08	0.09	67.65	2.0	2.0	1.6	2.1	1.7
Recleaner Concentrate 2	4.4	4.5	0.009	0.01	0.58	0.07	62.84	4.6	3.0	2.6	3.1	17.9	0.009	0.01	0.73	0.08	64.29	6.5	6.6	4.6	4.7	4.8
Recleaner Concentrate 3	16.2	16.7	0.009	0.02	0.54	0.09	46.44	17.0	14.7	8.8	14.5	48.8	0.009	0.02	0.59	0.09	51.44	23.2	23.6	19.3	13.5	19.3
Recleaner Concentrate 4	16.3	16.8	0.007	0.02	1.16	0.12	16.90	13.3	15.7	19.1	19.5	17.9	0.008	0.02	0.83	0.10	36.93	40.0	37.0	35.1	32.6	38.8
Recleaner Concentrate 5	16.8	17.3	0.005	0.02	1.02	0.10	1.92	9.8	14.3	17.3	16.8	2.1	0.007	0.02	0.89	0.10	26.35	57.3	46.8	49.4	50.0	55.6
Recleaner Concentrate 6	10.3	10.6	0.005	0.02	1.12	0.10	0.48	6.0	9.9	11.7	10.3	0.3	0.007	0.02	0.92	0.10	22.31	67.9	52.8	59.4	61.6	65.9
Recleaner Tailings	31.1	32.1	0.013	0.02	1.22	0.11	2.30	47.2	40.6	38.4	34.1	4.6	0.009	0.02	1.02	0.10	15.89	100.0	100.0	100.0	100.0	100.0
Feed	97	100.0	0.009	0.02	1.02	0.10	15.89	100.0	100.0	100.0	100.0	100.0										

13.2.9 Conclusions and Recommendations from the SGS UK Test Program

- Pre-concentration using dense media separation or conventional gravity separation at coarse sizes to reject a coarse waste product could be considered for tin recovery although the overall copper and zinc losses would be high. Heavy liquid test results demonstrated that with mass rejection from 25% to 51%, (mass rejection increasing with decreasing size from 11.2 mm to 1 mm) tin losses would be around 5-7%.
- The tin release curves give clear cut separations below 250 microns, with 40% mass pull to concentrate generating 97% tin recoveries. There is no clear separation for the tin until size is reduced to below 250 microns, which is an optimum size for further processing.
- Tin grades of over 5% were being achieved in all of the size fractions, albeit at low recoveries indicating the presence of free tin from the coarsest sizes downwards, which should be removed at the earliest opportunity to prevent over grinding of cassiterite as the grind size decreases.
- The magnetic separation testwork suggested that iron and sulphur could be removed magnetically, at low magnetic field strengths.
- Previous mineralogy investigations highlighted that pyrite/pyrrhotite and cassiterite particle size was typically coarse at circa 100 microns, with chalcopyrite and sphalerite particle size being finer at 50 microns. All of the minerals of interest were 60-80% liberated at +106 microns, which showed that a coarse bulk sulphide removal approach could be feasible.
- Gravity separation on bulk rougher flotation tailings at -200 microns produced a tin concentrate assaying 6.6% Sn at a stage recovery of 71.6%. Following further grinding of the 200-micron gravity tailings to 100 microns, a further gravity tin recovery of 9.7% was generated at a tin grade of 1.6% Sn giving a total recovery of 81.3%. Allowing for the 2% tin loss to sulphide concentrates, this generated an overall tin rougher recovery of 79.7%. Overall mass pull was circa 6-7%.
- Gravity separation following further milling of the 100-micron gravity tailings to 80% passing 50 microns generated a further tin stage recovery at a grade of 0.34% Sn from a head grade of 0.082% Sn. This represented a further overall tin recovery of 4%, bringing total gravity stage recovery to 85.5%.
- Tin rougher flotation on the 80% passing 50-micron gravity tailings gave higher tin recoveries than the gravity separation at between 63-67% at concentrate grades of circa 0.19-0.14% Sn. Mass pull to concentrate was high at approximately 25%. However, cleaning of the flotation tin concentrate using flotation proved to be difficult.
- Cleaning of the 200 and 100-micron rougher gravity concentrates produced tin stage recoveries of 85% and 77% respectively with tin concentrate grades of 14.2% Sn and 8.7% Sn, demonstrating that even at these coarser sizes, there is considerable tin liberation. The majority of the remaining tin was locked in the middlings, with only 5-9% of the tin lost to tailings.

- Gravity re-cleaning of the high intensity non-magnetic tailings generated tin grades of up to 63% Sn, albeit at a low stage recovery of 26%. A 75% tin stage recovery could be achieved at an acceptable tin grade of 50.5%.

After completing a variety of bench scale tests, SGS UK was able to develop a flowsheet for the East Kemptville deposit to produce copper, zinc and tin concentrates. Copper recovery was estimated at 86.4% into a 20.7% grade copper concentrate, zinc recovery was estimated at 84.5% into a 51.4% grade zinc concentrate and tin recovery into a 50.5% Sn concentrate was estimated at 76.8%. A detailed analysis of the final tin concentrate as produced by SGS UK is presented in Table 13.16.

Table 13.16
Final Tin Concentrate Analysis

Element	Sn	Cu	Zn	Fe	S	Pb	As	Cd
Value (%)	55.22	0.009	0.014	0.57	0.08	0.005	0.002	<0.0001
Element	**Ni**	**Co**	**Bi**	**Hg**	**Se**	**SiO_2**	**Mn**	**CaF_2**
Value (%)	0.006	<0.001	<0.0001	<0.0001	0.0001	9.04	0.35	0.55

13.3 MET-SOLVE LOW GRADE STOCKPILE TESTWORK

Using the SGS UK test results as a basis, Avalon contracted Met-Solve in Langley, BC (Met-Solve) in 2016 to undertake further flowsheet development testwork to recover tin from East Kemptville mineralization. The testwork program was divided into three phases:

- Phase I: Use of falcon gravity concentrators at 3 different grind sizes (200, 150 and 100 µm) to determine the sample's response to gravity concentration for the recovery of tin.
- Phase II: Grind material to 200 µm for the gravity rougher stage, followed by a regrind to 100 µm for gravity scavenging. Gravity tailings were then floated to attempt to recover additional tin.
- Phase III: Locked Cycle tests of the best flowsheet configuration previously identified in Phases I and II.

13.3.1 Metallurgical Sample

Avalon selected and shipped approximately 178 kg of crushed samples from the East Kemptville low-grade stockpile to Met-Solve. A composite was created from the combined samples which was analysed. In addition, head sample analyses were generated for a number of the tests the results of which are listed in Table 13.17. The average head grade for this sample (based on the various analyses) was 0.12% Sn which is higher than the direct assay of the composite sample at 0.10% Sn and also more in-line with previous stockpile sampling analyses. The inferred mineral resource grade of the stockpile is 0.11% Sn which compares well with the sample analyses.

Table 13.17
Tin Head Assay of Various Met-Solve Testwork Samples

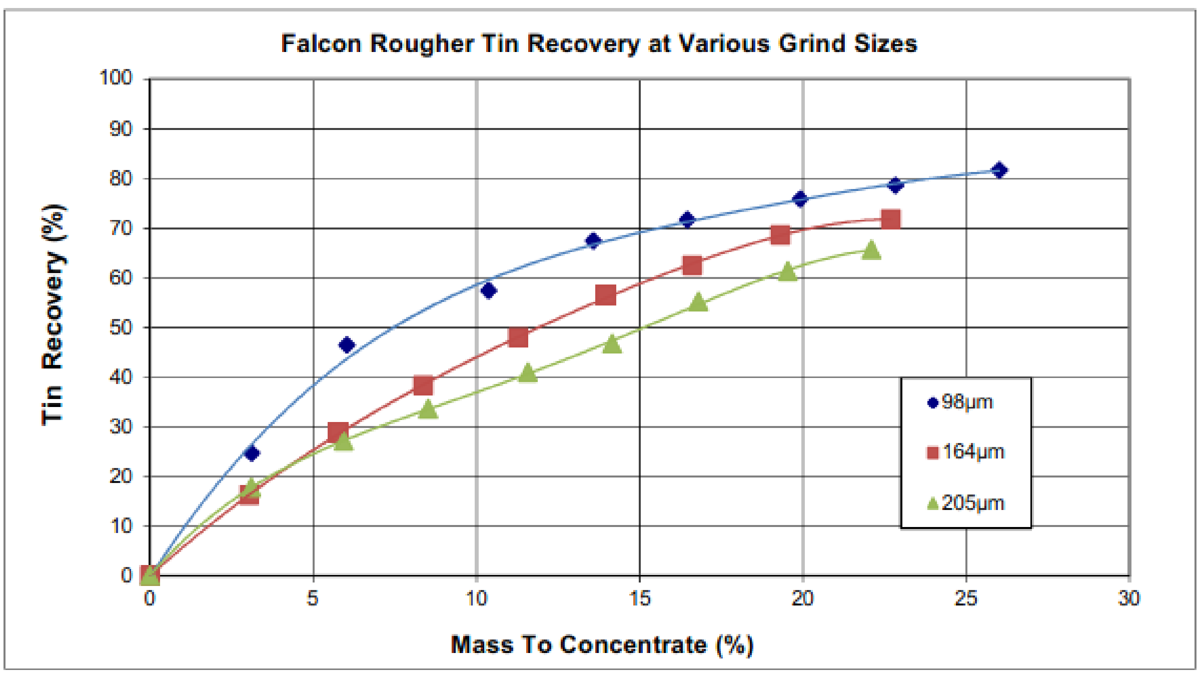
Test Description	Test Number	Sn (%)
SFA (Size Fraction Analysis)	WA101	0.13
Open Rougher Cleaner Test	WA102	0.12
Open Rougher Cleaner Test	WA103	0.12
Open Rougher Cleaner Test	WA104	0.12
Open Gravity-Float Test	WA105	0.12
Locked Cycle Test	WA201	0.11
Locked Cycle Test	WA301	0.11
Average Calculated Grade	-	0.12
Direct Assay	-	0.10

13.3.2 Phase I – Gravity Concentration

Results from Phase I found that gravity concentration was an effective method at recovering tin from the mineralized samples and grinding to 98 microns gave better results compared to the other coarser grind sizes. Figure 13.5 below shows the results of the three grinds used for the testwork.

The results indicate that the East Kemptville material responds well to gravity concentration, achieving up to 81.6% tin recovery with a 26% mass yield with the P_{80} grind of 98 microns.

Figure 13.5
Falcon Rougher Gravity Recoveries at Various Grind Sizes



Source: Met-Solve, MS1666 Avalon East Kemptville Gravity Testwork Report, October 2016

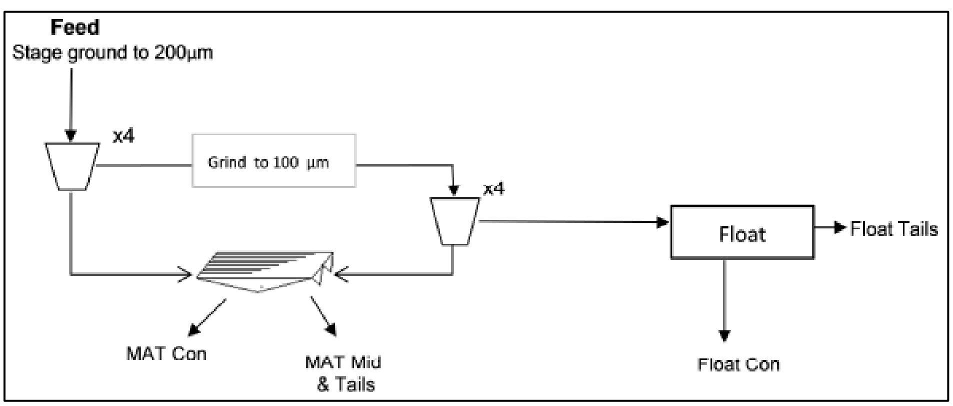
Additional testwork comprising rougher, cleaner, scavenger and cleaner scavenger gravity concentration using Falcons was completed. Cleaner gravity concentrate was further processed with a shaking table to improve tin grade. A final concentrate grading 45.8% Sn was produced albeit with poor tin recoveries due to the test being operated in open-circuit with no recycling of intermediate products (the scavenger tailings stream contained only 17.7% of the tin).

13.3.3 Phase II – Gravity and Flotation Testwork

Phase II of the testwork included a two-stage grinding circuit and a flotation circuit to recover tin from the scavenger gravity tailings stream. Overgrinding and slimes production can be a problem, so the first stage of gravity concentration was used to capture coarser tin minerals and a grind of 182 microns while the tailings from the first stage were re-ground to 109 micron and subjected to a second stage of gravity recovery. Figure 13.6 below illustrates the flowsheet used for the testwork.

Results from the Phase II testwork showed improved tin recoveries when combining flotation and shaking table concentrates, but the overall grade was low due to the poorer liberation from the coarser concentrate material. It was also shown that the intermediate grind at 100 microns had limited effect in improving recoveries as the bulk of the tin was recovered in the first/coarse stage. Although about 63.5% of the tin in the gravity tailings were recovered in the flotation concentrate, the potential to upgrade this material to a grade >40% was considered limited, and the cost of the additional flotation circuit would most likely outweigh any additional revenue.

Figure 13.6
Open Circuit Gravity-Flotation Flowsheet



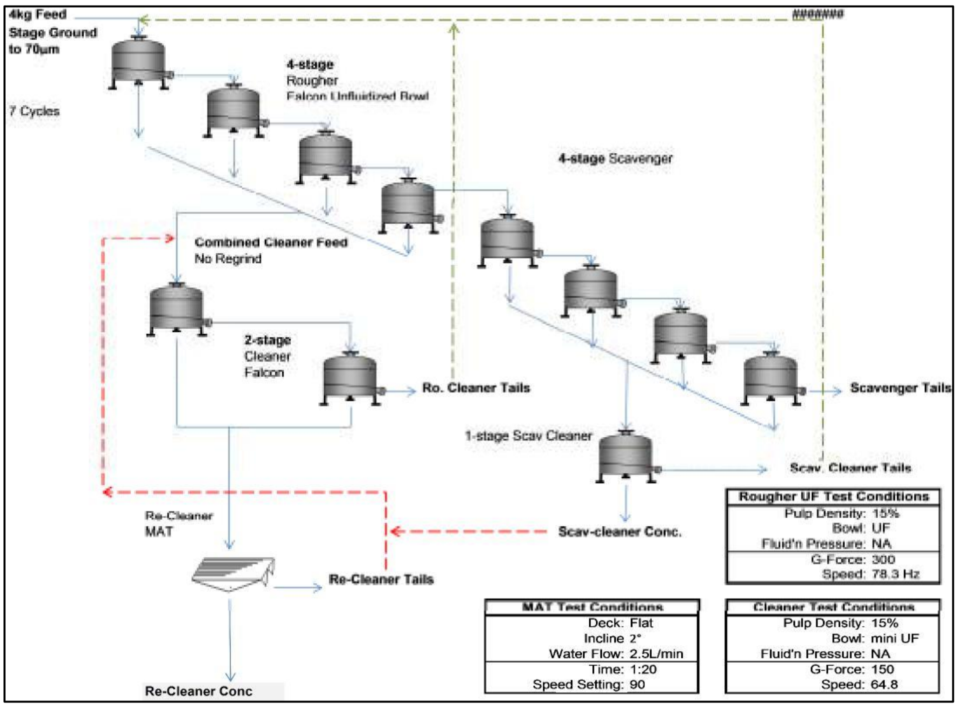
Source: Met-Solve, MS1666 Avalon East Kemptville Gravity Testwork Report, October 2016.

13.3.4 Phase III – Locked Cycle Tests

It was decided that doing a Locked Cycle Tests (LCT) was required to better determine the overall recovery potential, and that in order to keep the flowsheet to a single stage of grinding, a mill product size P_{80} of ~100 microns should be targeted.

The flowsheet used for Phase III of the testwork is presented in in Figure 13.7. It includes a rougher gravity circuit followed by a scavenger gravity circuit feeding a cleaner gravity circuit (with a cleaner scavenger stage) and finally a shaking table.

Figure 13.7
Phase III LCT Testwork Flowsheet



Source: Met-Solve, MS1666 Avalon East Kemptville Gravity Testwork Report, October 2016.

This work was conducted as a seven cycle LCT where tailings from each stage were recycled back to the front end. The initial test run was done with a feed grind P_{80} of 100 microns with a regrind of the rougher gravity concentrate to 70 microns prior to the cleaner stage, but tin recovery was poor in this case.

A second test was conducted with a feed grind of P_{80} = 70µm and with no regrind of the rougher gravity concentrate. Results from this test are included in Table 13.18.

, The tin concentrate grade produced by the second LCT was approximately 45% Sn with a 60% tin recovery based on the calculated head of 0.093% tin (which appears low compared to the previous assayed and calculated head analyses).

Analysis of the tin in the various recycle streams suggested that the 40.5% tin in the final tailings, contains 7.4% from the various recycle streams (i.e., scavenger gravity tailings alone contain only 33.1% of the plant feed tin content). This offers the potential to improve final recovery closer to 65% once a stable circuit is achieved and optimization of the recycle processes have been implemented.

Table 13.18
Locked Cycle Test 2 Results

Product	wt. %	Grade Sn (%)	F (%)	Sn Distribution (%)
Gravity Concentrate				
Re-Cleaner Con. Cycle 1	0.01	41.61	1.44	4.8
Re-Cleaner Con. Cycle 2	0.02	47.49	0.67	10.3
Re-Cleaner Con. Cycle 3	0.02	43.93	0.90	7.5
Re-Cleaner Con. Cycle 4	0.02	44.68	0.66	8.3
Re-Cleaner Con. Cycle 5	0.02	44.06	0.68	9.2
Re-Cleaner Con. Cycle 6	0.02	46.91	0.59	9.4
Re-Cleaner Con. Cycle 7	0.02	42.74	0.55	10.2
Total Re-Cleaner Concentrates	**0.12**	**44.67**		**59.5**
Average Con Grade from Last 3 Cycles		44.57	0.61	
Scavenger Tailings, Cycle 1	10.6	0.031		3.5
Scavenger Tailings, Cycle 2	14.0	0.036		5.4
Scavenger Tailings, Cycle 3	15.7	0.040		6.7
Scavenger Tailings, Cycle 4	14.7	0.038		5.9
Scavenger Tailings, Cycle 5	15.3	0.041		6.6
Scavenger Tailings, Cycle 6	14.1	0.038		5.8
Scavenger Tailings, Cycle 7	15.5	0.040		6.6
Total Bulk Gravity Tailings	99.9	0.038		40.5
Calculated Head from LCT	**100.0**	**0.093**		**100.0**
Assayed Head		0.100		
SFA Calculated Head		0.104		

13.3.5 Concentrate Upgrading by Flotation to Remove Base Metal Sulphides

The tin concentrate produced by the LCT discussed above contained over 20% sulphides, predominantly iron sulphides. An investigation was conducted to remove these to produce a higher-grade tin concentrate without significant tin losses.

To further upgrade the final tin concentrate, Met-Solve conducted further testwork investigating the use of froth flotation together with low intensity magnetic separation (LIMS) to remove sulphide material from the Locked Cycle concentrate. The flowsheet used was the same as Figure 13.7 with the addition of sulphide flotation and magnetic separation between the Falcon cleaners and the final shaking table cleaner. The feed to this test circuit was ground to a P_{80} of 79 µm.

The LIMS and sulphide float removed ~60% of the iron and 93% of the sulphur present. The final tin concentrate after removal of these increased to 68.3% tin with only very minor losses of tin (approximately 0.2% losses to magnetics and 1.4% losses to the flotation concentrate). It is believed that much of these losses could be recovered by subjecting both the magnetics and float concentrate to cleaner operations as any tin present is most likely the result of physical entrainment in the material being removed.

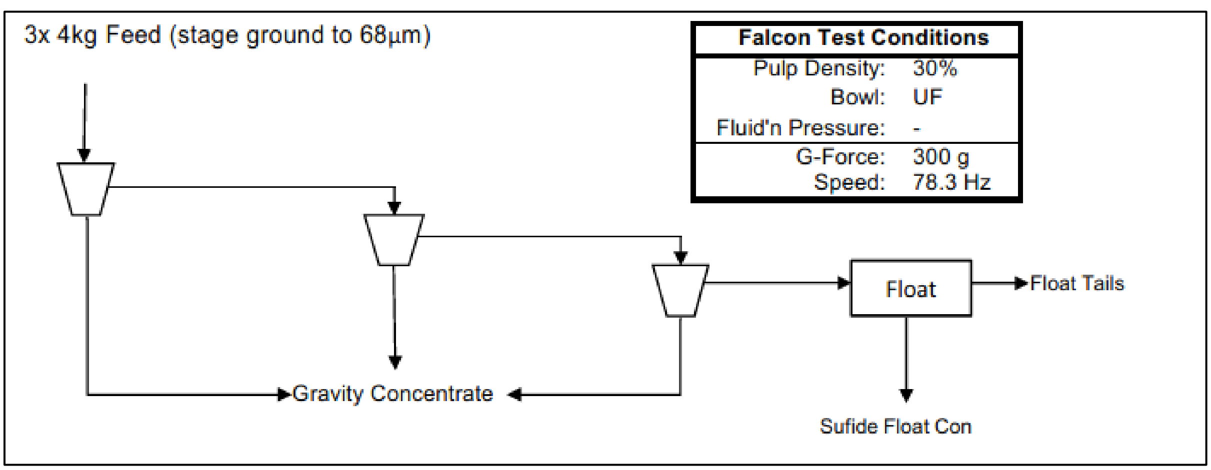
The final tin concentrate grade of >68% is far greater than the targeted 55% concentrate. This suggests that it is reasonable to assume that the eventual tin recovery will be higher than the 58-59.5% achieved in these tests when producing the lower grade. product.

13.3.6 Sulphide Removal from Gravity Tailings

Following the development work covered in Section 13.3.3 and 13.3.4, Met-Solve performed further investigations to look at using sulphide flotation to reduce sulphide levels in the gravity tailings to below 0.1% wt. % sulphur such that they could be used as capping material for sealing the tailings dam. The scope of the work included producing a bulk composite for gravity tailings (after tin removal) following which, the sample was subjected to sulphide flotation to reduce the suphide grade. Met-Solve conducted a number of scoping flotation tests with SIPX and PAX reagents at different dosages to achieve the low sulphide requirement for the tailings.

Avalon sent approximately 167 kg of Main Zone drill core sample to Met-Solve in April, 2018. Met-Solve then crushed the material to 2 mm before stage-grinding the sample to a grind size of 68 µm. The calculated head sample contained 0.15% Sn and 0.62% S. The milled material was subjected to a two-stage gravity process with the rougher gravity tailings from the first stage feeding the second gravity process. Concentrates from the two stages were collected while the rougher scavenger gravity tailings were used for sulphide flotation testing. Figure 13.8 below shows the flowsheet and Falcon concentrator test conditions.

Figure 13.8
Met-Solve Gravity Tailings Sulphide Removal Flowsheet



The results from this test are summarized in Table 13.19.

Table 13.19
Summary of Results from Met-Solve Gravity Tailings Sulphide Removal Test

Products	Weight (%)	Assay (%)		Distribution (%)	
		Sn	S	Sn	S
Falcon Concentrate	22.37	0.54	1.13	75.6	40.4
Rougher + Scav Float Conc	4.27	0.09	8.30	2.4	56.6
Scavenger Float Tails	73.36	0.05	0.025	22.0	2.9
Calculated Head	100.0	0.16	0.63	100.0	100.0
Assay Head		0.16	0.64		

The sulphide flotation stage was able to reduce the levels of the sulphur in the scavenger tailings to 0.025% S, which is well below the target of 0.1% S. In reviewing the results, it was noted that the sulphur content of the rougher tailings (scavenger feed) was 0.05% which suggests that the scavenger flotation stages would not be required. By eliminating the second sulphide rougher and the rougher scavenger, PAX addition would be reduced to 95 g/t and neither copper sulphate or Lime (for pH control) would not be required.

13.4 CONCLUSIONS

The testwork on samples representing the East Kemptville mineral resources has resulted in the development of a flowsheet capable of producing a tin concentrate. The material will be milled to P_{80} ±80 microns before being put through a series of Falcon gravity concentrators. Falcon concentrates will feed a magnetic separation circuit followed by a simple sulphide flotation circuit to remove sulphides. The non-sulphide flotation tailings will be cleaned using final shaking table gravity circuit the concentrates from which will be collected and dewatered before being shipped to potential customers.

In order to minimize over-grinding of the cassiterite a gravity recovery stage will be installed within the milling/classification process which will reduce the recirculation of liberated tin to the ball mill.

The recoveries and product grades used in the plant design, based on the testwork described above, are presented in Table 13.20.

Table 13.20
Basic Process Design Criteria from Metallurgical Testwork

Parameter	Units	Value
Grind Size P_{80}	Microns	80
Tin losses to sulphide/magnetics concentrate	% In Mill Feed	1.6
Tin recovery to final concentrate	%	60
Grade of the tin gravity concentrate	% Tin	55

The tailings from the gravity circuit can be treated through a bulk sulphide flotation process to reduce contained Sulphur content to approximately 0.05% S, making it a suitable material for capping of the tailings dam.

14.0 MINERAL RESOURCE ESTIMATES

An updated mineral resource estimate for the East Kemptville Project was completed on 7 May, 2018 (see Avalon news release dated 28 June, 2018). The mineral resource estimate is based on a block model prepared by Avalon and is summarized in Table 14.1. The deposit was subdivided into the Main Zone and the Baby Zone, which were interpolated separately. The in situ unmined tin resources were estimated using historic drill holes, data from drill holes completed by Avalon in 2014 and 2015, a post-mining topographic model, and constrained within a Whittle pit shell. A tin cut-off grade of 0.10% was considered as reasonable based on current mine plans and historic cut-off grade used at the East Kemptville mine.

Table 14.1
Updated Mineral Resource Estimate for the Main and Baby Zones
(Dated 7 May, 2018)

Classification	Cut-off grade Sn (%)	Main Zone NE		Baby Zone		Total	
		Tonnes (Mt)	Sn (%)	Tonnes (Mt)	Sn (%)	Tonnes (Mt)	Sn (%)
Measured	0.08	0.40	0.173	0.22	0.241	0.61	0.197
	0.10	**0.38**	**0.177**	**0.20**	**0.251**	**0.58**	**0.203**
	0.12	0.32	0.188	0.19	0.259	0.51	0.214
Indicated	0.08	27.89	0.133	1.72	0.194	29.61	0.137
	0.10	**20.91**	**0.148**	**1.48**	**0.211**	**22.39**	**0.152**
	0.12	14.84	0.163	1.27	0.228	16.11	0.168
Measured + Indicated	0.08	28.28	0.134	1.93	0.199	30.22	0.138
	0.10	**21.29**	**0.148**	**1.68**	**0.216**	**22.97**	**0.153**
	0.12	15.16	0.164	1.46	0.232	16.62	0.170
Inferred	0.08	18.54	0.125	0.90	0.153	19.43	0.126
	0.10	**13.56**	**0.137**	**0.69**	**0.172**	**14.25**	**0.139**
	0.12	8.11	0.156	0.51	0.193	8.62	0.158

Notes:
1. CIM Definition Standards for Mineral Resources, 2014, were followed.
2. The Qualified Person for this Mineral Resource estimate is William Mercer, Ph.D., P. Geo. (Nova Scotia). The mineral resources are current as of May 7, 2018.
3. The mineral resource estimate is based on 194 drill holes totalling 21,456 m drilled between 1979 and 1991 by previous operators and 23 holes totalling 4190 m drilled by Avalon in 2014 and 2015.
4. Drill data were organized in Maxwell DataShed and for estimation purposes were transferred to the Geovia GEMS 6.8.1 software, wherein the block model was developed.
5. Resources were estimated by interpolating composites within block models of 24 m by 24 m by 12 m blocks in the Main Zone and 6 m by 6 m by 6 m in the Baby Zone. Interpolation used the inverse Ordinary Kriging method.
6. In the Main Zone, Measured material was defined as blocks interpolated with a search ellipse with radii of 40x20x15 m using 18-36 samples, corresponding to 3-6 drill holes, indicated material with a 120x40x18 m search ellipse and the same number of samples, and inferred material with a 315x85x18 m search ellipse using 12-24 samples corresponding to 2-4 drill holes. In the Baby Zone, Measured material was defined as blocks interpolated with a search ellipse with radii of 30x20x8 m using 6-12 samples, corresponding to 3-6 drill holes, indicated material with a 48x33x12 m search ellipse and the same number of samples, and inferred material with a 95x65x24 m search ellipse using 4-8 samples corresponding to 2-4 drill holes (see Section 1.12 Resource Classification).
7. Prior to compositing, the assays were capped at 1% Sn, which corresponds to the 99th percentile of the tin assay data, reducing the length-weighted mean of the tin assays by 9.4%.
8. Mean density values of available data of 2.728 t/m^3 and 2.784 t/m^3 were used for the Main and Baby Zones, respectively
9. The resource estimate has been constrained using the Whittle pit described previously (Avalon News Release 15-02, February 25, 2015).

10. Several possible cut-off grades are reported in this resource estimate. Based on past mining practice at East Kemptville, a cut-off grade of 0.1% Sn is reasonable and preliminary cost and revenue values at the time of estimation also suggest this is reasonable.
11. Mineral resources do not have demonstrated economic viability and their value may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other issues.

The Qualified Person (QP) for the Baby and Main Zone mineral resources reported in the PEA is William Mercer, P. Geo. who is not independent from Avalon. However, these current resource estimates have not changed significantly since the previous versions that were prepared independently by Hains Engineering with its principal, Donald Hains (P. Geo), serving as the independent QP for the purpose of NI 43-101 (News Release 14-13, October 31, 2014).

There has been no material change of the mineral resource estimate for the Low Grade Stockpile since the previous estimate by Hains with an effective date of 16 November, 2015 (see Annual Information Form 20F, Avalon Advanced Materials, August 31, 2016, Table EK2, Page 82, accessible on SEDAR (www.sedar.com) and Avalon's website under Financial Statements 2016 at the following link: http://avalonadvancedmaterials.com/investors/regulatory_filings/).

Table 14.2
Low Grade Stockpile Estimated Inferred Mineral Resource

Category	Tonnes (Mt)	Grade (%)		
		Sn	Zn	Cu
Inferred	5.87	0.112	0.100	0.61

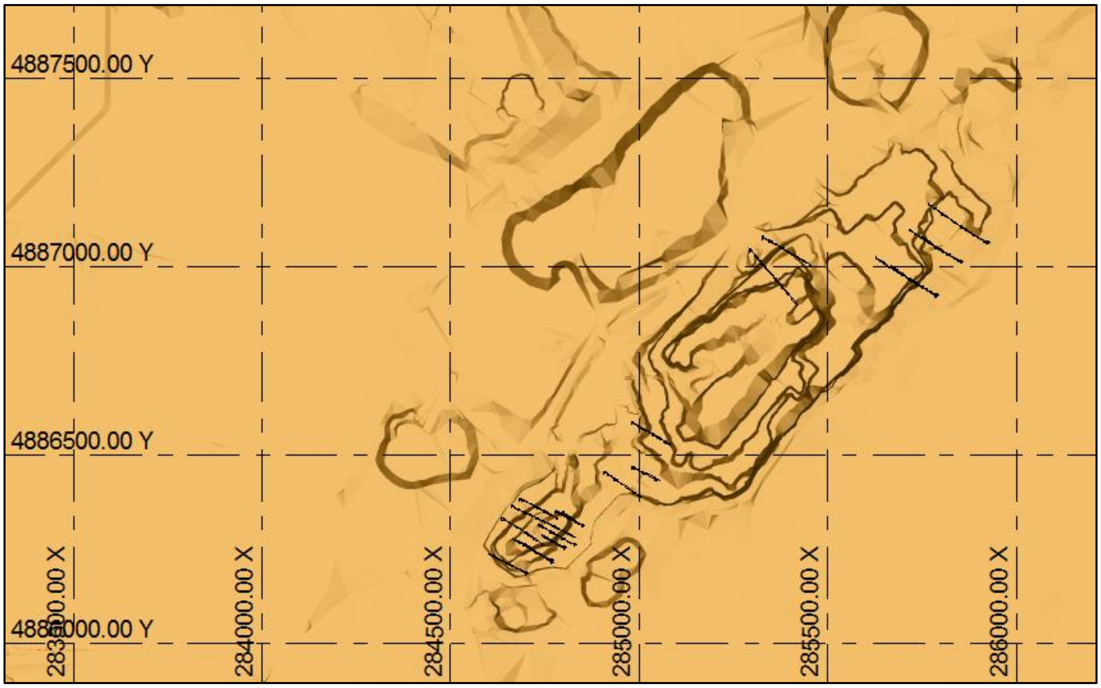
Notes:

1. This estimate is as of 16 November 2015.
2. CIM Definition Standards 2014 were followed for mineral resources.
3. The independent Qualified Person for this Mineral Resource estimate is Donald Hains, P.Geo., of Hains Engineering Company Limited.
4. Resources were estimated by examination of historical RAL data and Avalon's 2015 sampling of the Low-Grade Stockpile.
5. Mineral resources do not have demonstrated economic viability and their value may be materially affected by environmental, permitting, legal, title, socio-political, marketing or other issues.

14.1 MINERAL RESOURCE DATABASE

Avalon's Maxwell Datashed database for the May 7, 2018 mineral resource estimate for the East Kemptville Project contained 472 drill holes with a total length of 62,893 m. The Avalon database is based on the database of historical East Kemptville data built by a Nova Scotia geological consultant using original drill hole logs and assay reports. The historical database was imported to Avalon's Maxwell Datashed database and verified. The database contains 443 historical holes (57,395 m) drilled from 1979 to 1991 first by Shell and then by Rio Algom and 29 drill holes (5,498 m) drilled by Avalon between 2014 to 2015 (Figure 14.1) to verify and expand on the historical drilling. The database includes collar, downhole survey, lithology and assay tables.

Figure 14.1
Plan Showing Post-Mining (1992) Topography with the Traces of the Holes Drilled by Avalon Between 2014 and 2015 Projected to the Surface.



The assay data set in the Maxwell Datashed database includes 15,309 drill hole intervals, of which 3,093 are from the 2014 and 2015 drill holes. Of these intervals, 14,277 contain Sn assay values, 12,363 have Zn and 12,232 have Cu assay values.

An inspection of the data and the available original assay sheets indicated that for the missing assay values, the data are not available because the original paper copies of the assay sheets were lost. The missing intervals were thus removed prior to processing to avoid the generation of artificial zero-grade assay intervals.

Not all drill holes in the project database are located within the resource model or have valid collar data (Figure 14.2). The data was imported into GEMS 6.8.1 for geological modeling and resource estimation. During the import into GEMS, 222 samples were discarded because they are duplicates. Following removal of the empty Sn assay intervals and import into GEMS, the sample intervals have a median length of 3.00 m and an average of 2.49 ± 0.92 m (1σ) both in the overall data set (14,055 samples) and in the subset for the modeled granite geology 3D-solid (9,178 samples). Table 14.2 summarizes the assay statistics for the project database including all available data.

Figure 14.2
Plan Showing the 1992 Topographic Model and the Traces of All Drill Holes in the Database

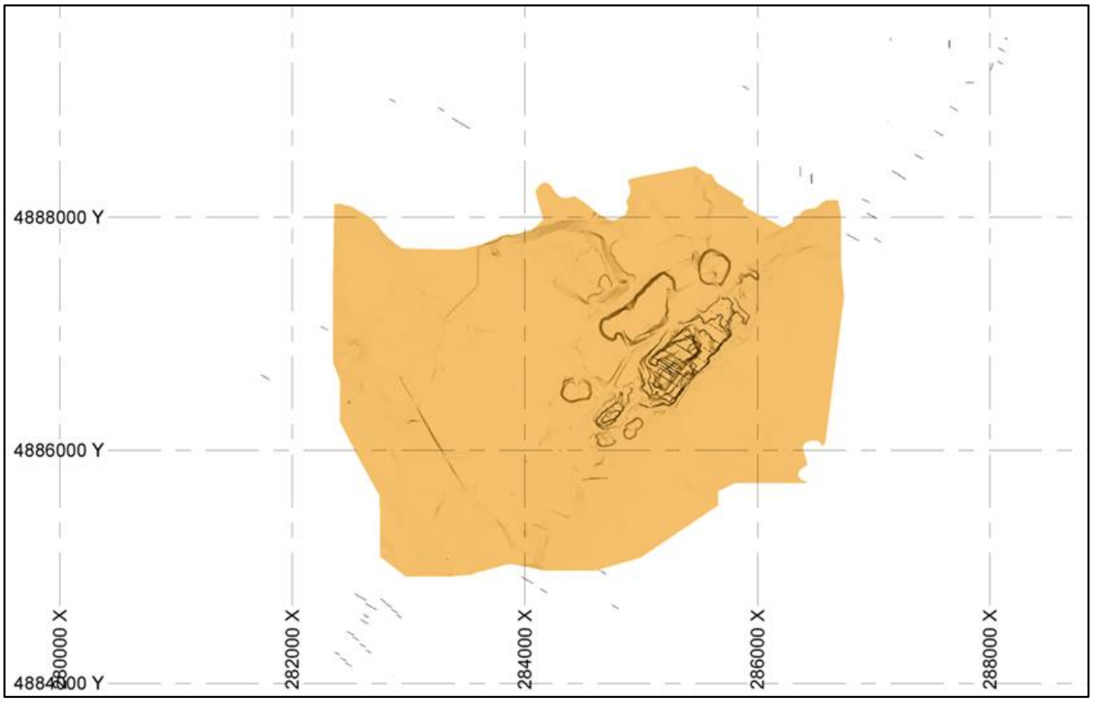


Table 14.3
Assay Statistics – East Kemptville Project Database

All Assays	Sn	Zn	Cu
Count (N)*	14,277	12,363	12,232
Minimum (%)	0.000	0.000	0.000
Maximum (%)	27.20	7.14	3.00
Mean (%)	0.110	0.129	0.050
Median (%)	0.032	0.056	0.021
Standard Deviation	0.493	0.227	0.097

The density database table includes data for historic holes (Wardrop, 2010) and for the holes drilled by Avalon in 2014 and 2015. It contains 2,954 density entries, ranging from 1.680 to 7.970 g/cm^3 with a mean of 2.751 g/cm^3 and a median of 2.72 g/cm^3. The density data is further analysed in Section 14.6 including identification of erroneous data.

For this study, all available 3D coordinate data was transformed into the UTM NAD83 Zone 20 N and CGVD2013 elevation systems using Datashed transformations and transformations in GEMS. It should be noted that the original database used NAD27 UTM coordinates as the data was already prepared in this format by a Nova Scotia consultant. Present general practice in Nova Scotia is to utilize NAD83 UTM coordinates.

In addition, the original mine elevations are 21.635 m higher than NAD83 CGVD2013 elevations. The calculation of this difference is based on survey of an iron bar on the East

Kemptville site with an original mine site elevation of 87.837 m. The NAD 83, CGVD2013 elevation of this same point is 66.202 m. Thus, the historic mine-based elevations are 21.635 m higher than NAD83.

14.2 DATABASE VALIDATION

The data exported from the Avalon database were validated by Avalon staff directed by William Mercer P.Geo, and the procedures are described in Sections 11.0 and 12.0 detailing the data verification and quality assurance/quality control procedures. The data imported into GEMS was randomly validated against the original drill hole logs and assay reports. The location of holes is reliant on historic data from qualified land surveyors as reported by Rio Algom for the historical holes and reliant on Nova Scotia registered land surveyors for the 2014 and 2015 drill holes.

14.3 GEOLOGICAL INTERPRETATION

The East Kemptville deposit is a greisen deposit in which the four major lithologies are recognized:

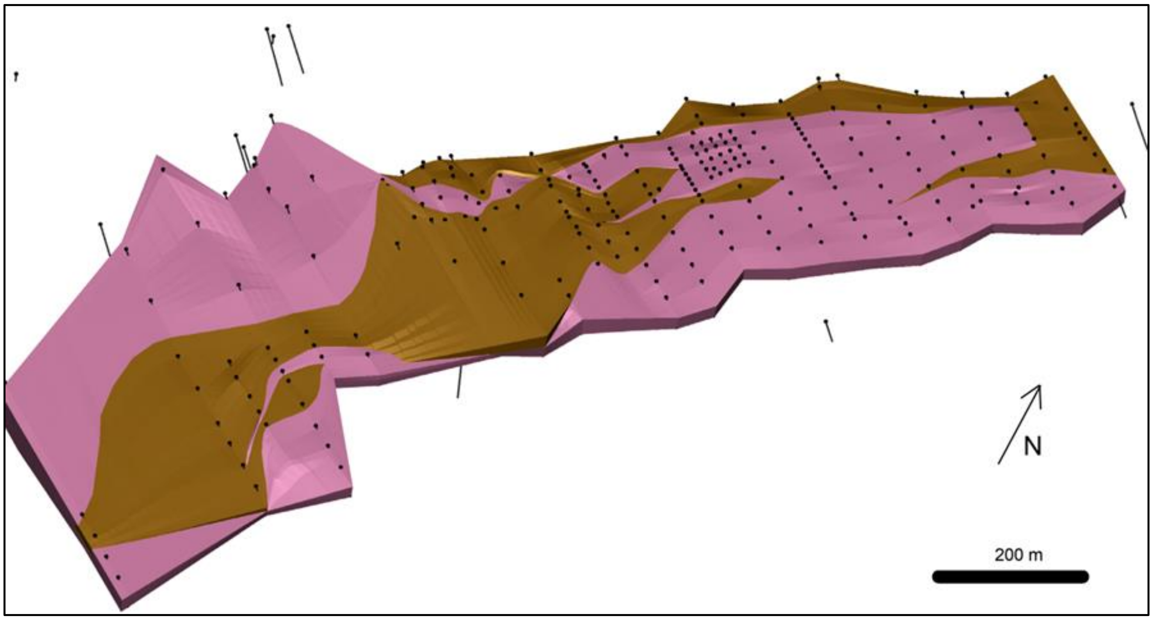
1. Granite intrusive, which hosts most of the mineralization.
2. Contact zone, which is a mineralized breccia containing both metasediment and granite clasts.
3. Metasediments.
4. Overburden.

The greisen veins generally strike 030° and dip steeply and the geometry of the contact between the granite and the metasediment is the main control on mineralization (Rio Algom, 1983).

Three geological units were modeled using the GEMS 6.8.1 3D software based on the geology lithology table of the database: granite, metasediment and overburden. Although the granite in the Baby Zone possibly represents a separate intrusion, it was found to be geologically continuous with the granite of the Main Zone and the granite further to the southwest. The contact zone, described as a mixture of granite and metasediment in drill logs, was found to be discontinuous in 3D-space and was thus merged with either granite or metasediment on a majority basis according to the drill logs. The 3D outline of the granite was used as the limit of mineralized material in the resource estimate. In the block model, blocks below the topography which were not within either of the geology 3D solids, were assigned the rock code 'unknown'.

Although the lithological contacts were interpreted accurately, faults have not yet been interpreted in the 3D model, mainly due to a lack of data providing their geo-referenced locations. Future resource estimates may be improved through interpretation of the faults crossing the deposit.

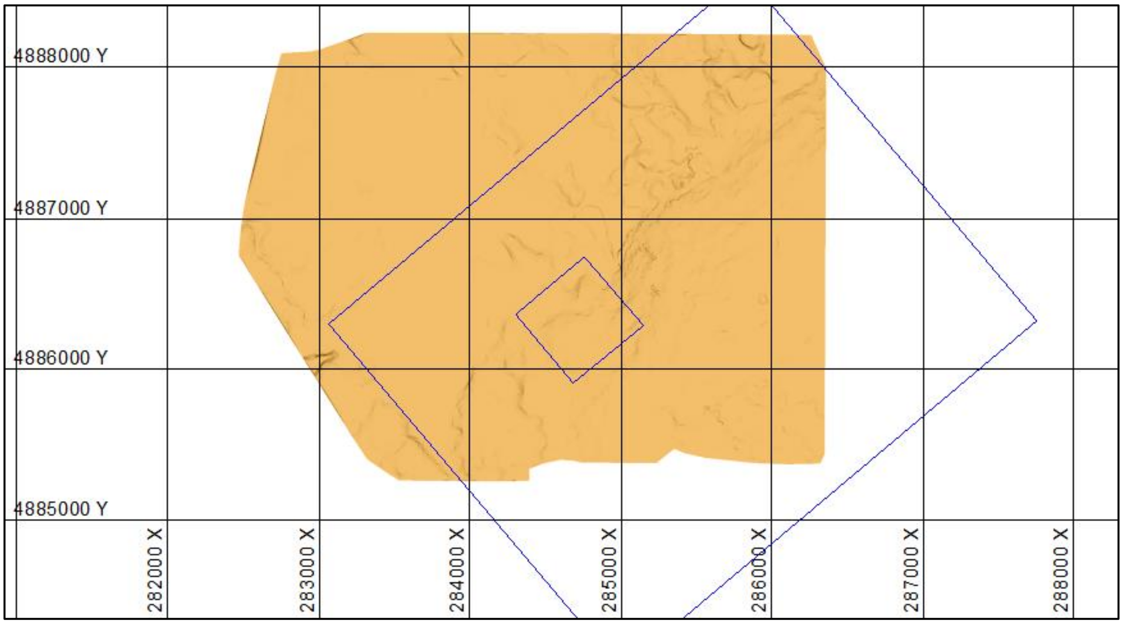
Figure 14.3
Three-Dimensional View Showing the 3D Wireframes for the Granite (Pink) and the Metasediment (Brown) – Both Pre-Mining and the Drill Hole Traces



14.4 TOPOGRAPHY

The model of the original pre-mining topography in 1983 (Figure 14.4) is based on a 1:5,000 scale site plan from the 1983 Nova Scotia Department of Natural Resources Property Report PR_ME_1983-010. The map was scanned and registered using the grid lines on the map. Contour lines were digitized, and the surface was modeled from this data in Surpac software. The model correlates well with the available surveyed drill hole collars with an average difference of less than 0.1 m.

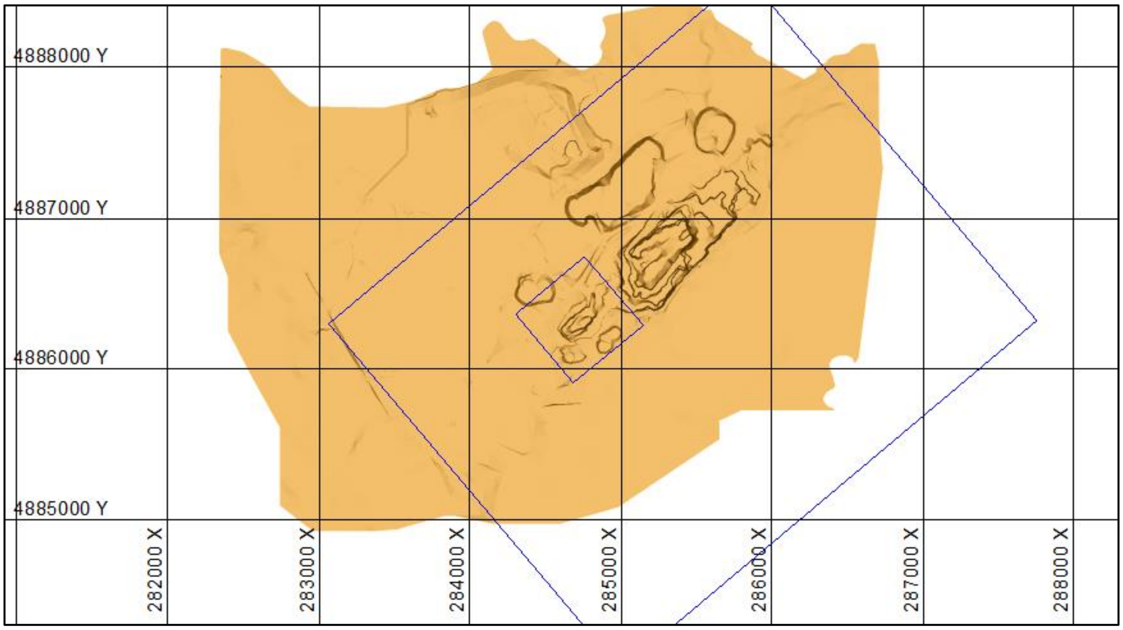
Figure 14.4
1983 Digital Topographic Model with Resource Model Boundaries



The model of post-mining topography in 1992 (Figure 14.5) is based on a compilation of eighteen 1:1,000 scale final survey maps by the Nova Scotia Department of Natural Resources.

A minimum of 15 control points were added to each map and then contour, berm, toe, feature lines, and spot elevations were digitized using appropriate elevations. The surface was modeled from this data in Surpac software. Random checks on control points and grid lines on the original maps and the overlain digitized files generally indicate an accuracy of +1.0 m in plan view.

Figure 14.5
1992 Digital Topographic Model with Resource Model Boundaries



14.5 SUBDIVISION OF ZONES IN THE DEPOSIT

Whereas the tin-mineralized granite body is continuous along strike, the historic mine operated in two pits, namely the Main and the Baby pits (Figure 14.6). They are located in the northeastern part of the known occurrence of the granite body (Figure 14.7). For these two areas, hereafter referred to as the Main Zone and the Baby Zone, the drilling coverage is considerably more systematic and closely spaced than outside them and the drill spacing is significantly closer in the Baby Zone than in the Main Zone (Figure 14.7). These two zones were thus used as a subdivision for all data for the purpose of this resource estimate. A third zone, which has been explored historically and by Avalon, the Duck Pond Zone, is not part of the current resource estimate.

Figure 14.6
Post-Mining Topography (1992) with the Two Open Pits, Both Currently Flooded

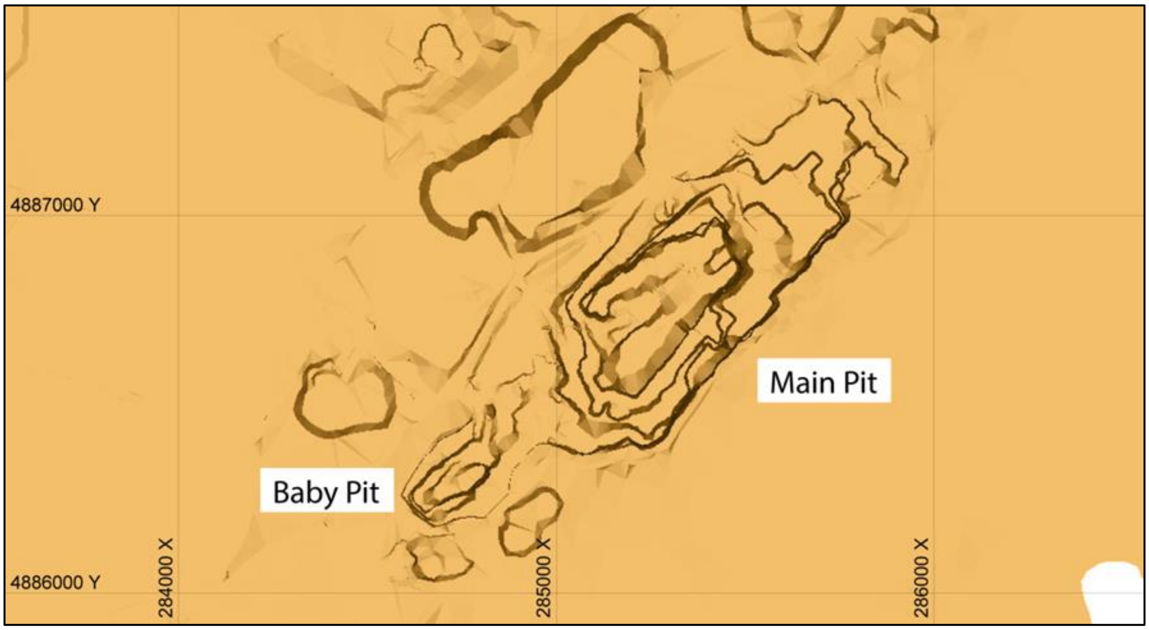
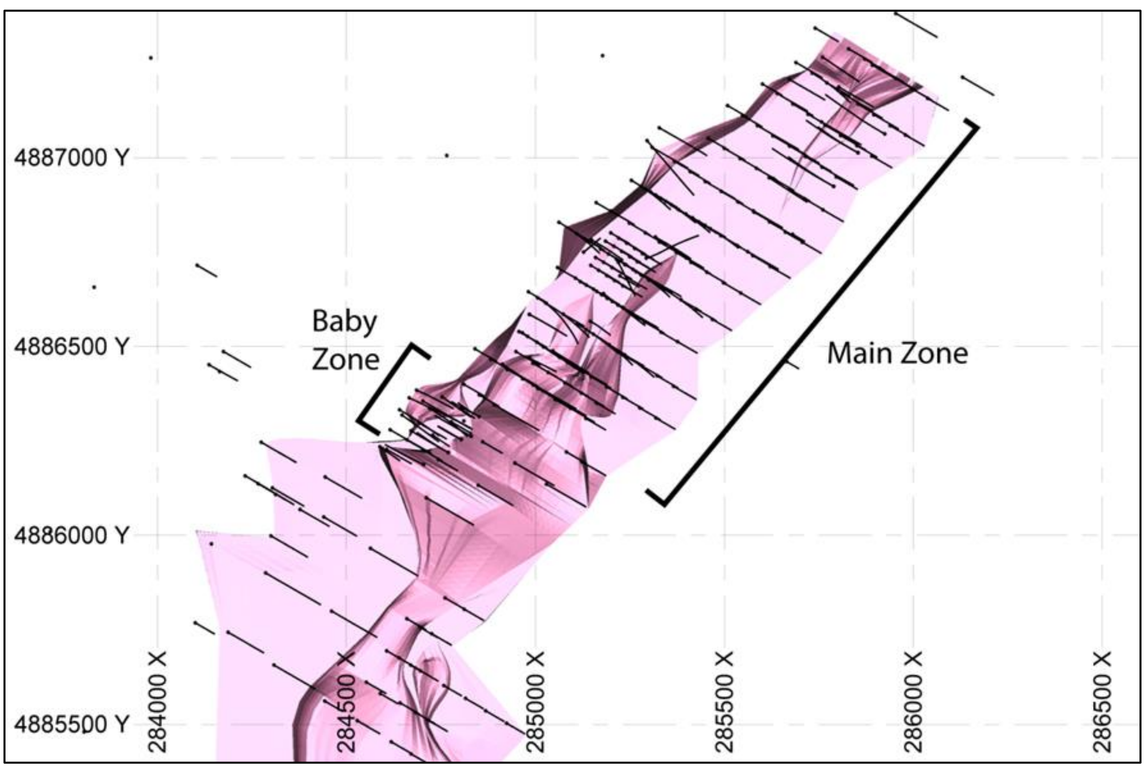


Figure 14.7
The Pre-Mining Granite 3D Wireframe and the Drill Hole Traces Projected to the Surface



14.6 ROCK DENSITY

Wardrop (2010) reported a density of 2.737 ± 0.101 t/m^3 (1σ, N = 1,107) for the granite in the Main Zone, a density of granite in the Baby Zone of 2.747 ± 0.096 t/m^3 (N = 83) and a density of the metasediment of 2.699 ± 0.034 t/m^3 (N = 7). Rio Algom's 1983, feasibility study reported a density of the run-of-mine mill feed of 2.75 t/m^3. The feasibility study also reports additional density data with an average of 2.79 ± 0.08 t/m^3 (N = 70, mineralized BQ core). The 2014 resource estimate prepared by B. Webb used a median granite density of 2.78 t/m^3 for all mineralized material (N=184, using only Avalon's 2014 data). (ref. Avalon News Release 14-13, October 31, 2014).

The density estimates for the present resource calculation uses a filtered subset (1,996 values) of all available data (N = 2,954) from Avalon's Datashed database, including the measurements of Wardop (2009) on historic core and those on Avalon's 2014 and 2015 diamond drill holes. Following a detailed inspection of the full database, the density data for holes EKAV-15-008, EKAV-15-009, EKAV-15-016, EKAV-15-017, EKAV-15-018, DPAV-021 to DPAV -024, and EKAV-15-025 to EKAV-15-029 were removed from the data set, because they contain unreasonably high densities which are not explained by any geological observations noted in the drill logs. This suggests a series of methodically flawed measurements; it is recommended that the density measurements for these holes are verified and/or re-acquired. This filtering removes most of the data for the Duck Pond area, which is not part of the present resource estimate. A further five outliers with densities of 1.68, 1.97, 3.31, 3.37 and 3.45 t/m^3 were removed from the data set for the Main Zone granite because they fell outside of the main data population.

As the 3D-coverage is insufficient for an interpolation of the density, the current resource estimate uses mean values. Separate density values for the granite in the Main and Baby Zones were calculated in accordance with reported petrological differences between the two granites (Figure 14.8, Table 14.4). Although their density ranges overlap within the standard deviations, the granite in the Main Zone (2.728 t/m^3) is slightly lighter than that in the Baby Zone (2.784 t/m^3, Table 14.16). These values are comparable to those reported historically for the mineralized granite (2.75-2.79 t/m^3; Rio Algom, 1983). In the Main Zone, only a small group of very high-grade samples displays a correlation with Sn grade. In the Baby Zone, a weak correlation between density and Sn grade and stronger correlations with Zn and Cu were observed. The Contact Zone lithology of the Baby Zone (not assigned for the resource estimate) has an average density of 2.815 t/m^3, possibly owing to the presence of tourmaline.

Figure 14.8
Histograms for the Density of the Granite in the Main and Baby Zones

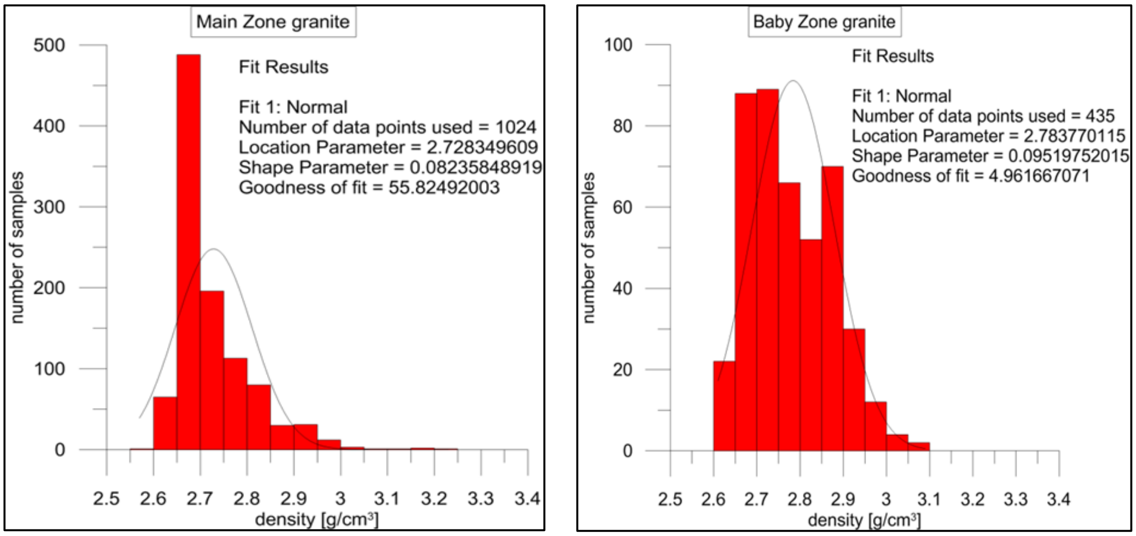


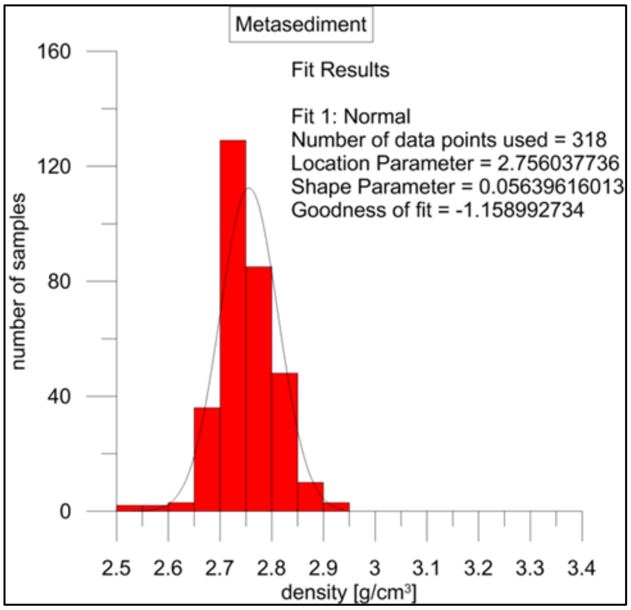
Table 14.4
Density Measurement Statistics

Description	Granite		Metasediment
	Main Zone	Baby Zone	
Number of samples	1024	435	318
Minimum (t/m^3)	2.57	2.61	2.53
Maximum (t/m^3)	3.23	3.09	2.94
Average (t/m^3)	2.728	2.784	2.756
Standard deviation (t/m^3)	0.082	0.095	0.056
Median (t/m^3)	2.7	2.77	2.75
Mode (t/m^3)	2.67	2.68	2.71, 2.73

For the metasediments, no difference in density was observed between the Main and Baby Zones, hence a single value of 2.756 t/m^3 was used (Table 14.4, Figure 14.9).

As no empirical measurements are available, the overburden was assigned a density of 1.8 t/m^3, a value based on experience and literature review.

Figure 14.9
Combined Histogram for the Density of the Metasediment in the Main and Baby Zones



14.7 TIN ASSAY STATISTICS

After import into the GEMS software, the database contains 14,055 assay values for tin ranging from 0.0000 to 27.2 wt. % Sn with a mean of 0.1084 wt. % Sn, a median of 0.0310 wt. % Sn and a standard deviation of 0.4886 wt. % Sn (Table 14.5). The data set is extremely skewed by relatively few high-grade analyses (skewness is 30.1, see Figure 14.10).

Table 14.5
Statistics for Tin Assays in the Entire Database and in the Subset of Intervals within the Granite Geology Solid

Statistic Description	All data (N = 14,055)	Granite Geology Solid (N = 9,178)
	wt. % Sn	wt. % Sn
Minimum	0.0000	0.0000
Maximum	27.2000	27.2000
Mean	0.1084	0.1419
Standard deviation	0.4886	0.5921
Length-weighted mean	0.0906	0.1173

Approximately 99.8% of the samples are <3 wt. % Sn and the 99.0 percentile correspond to approximately 1 wt. % Sn (Figure 14.11).

The very high-grade (above approximately 1 wt. % Sn) samples are scattered and not concentrated in a particular localized volume of the deposit.

Figure 14.10
Combined Histogram for the Tin Assay Data Limited to a Maximum of 3 wt. % Sn

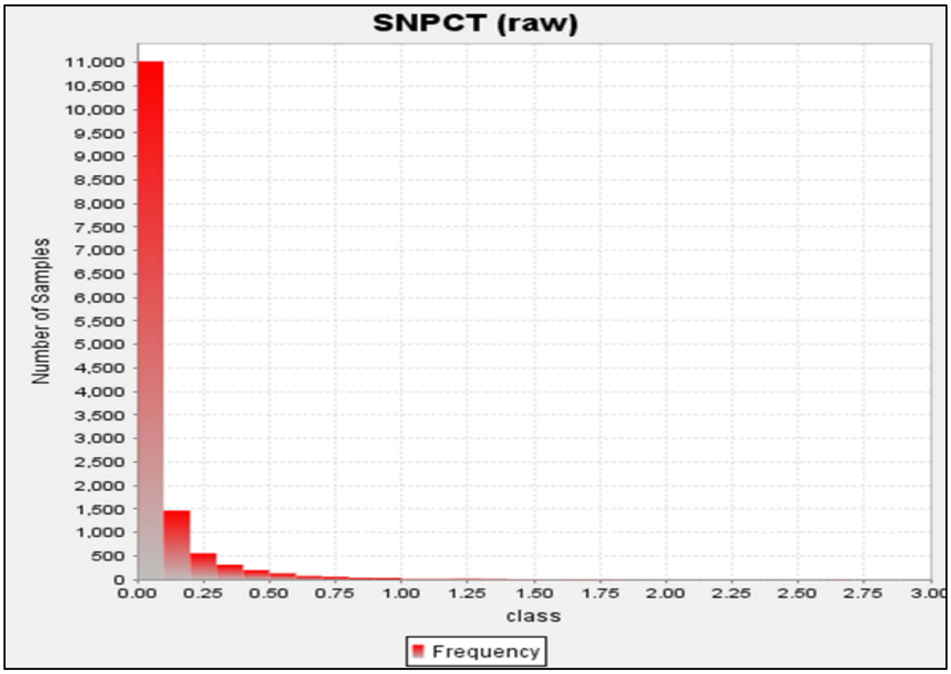
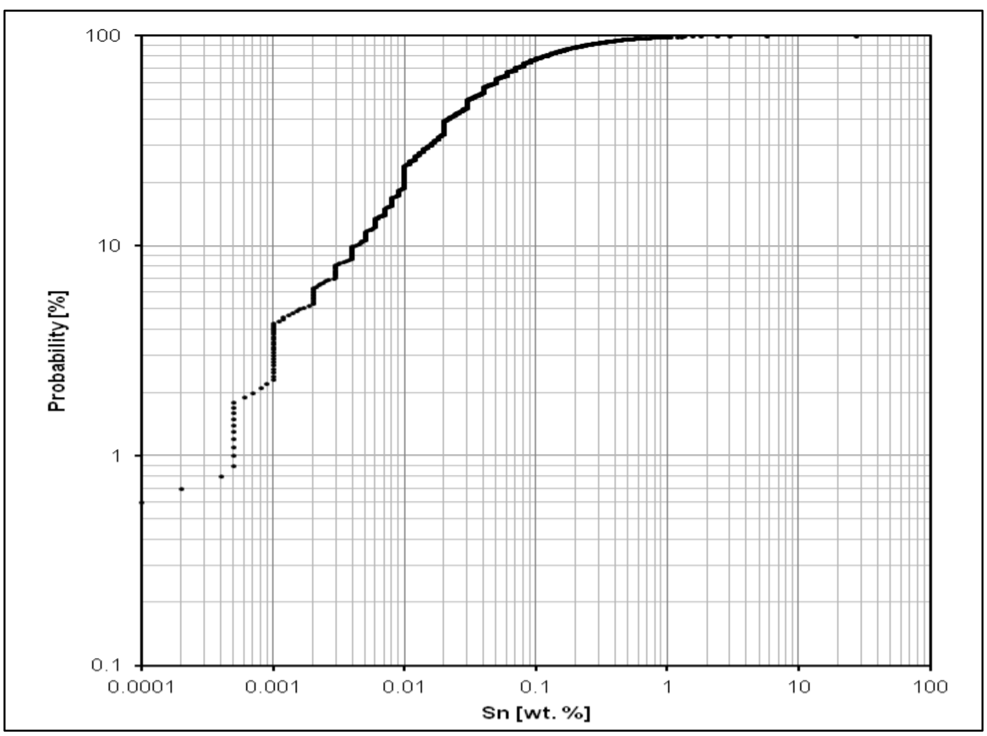


Figure 14.11
Cumulative Probability Plot for the Sn Assays



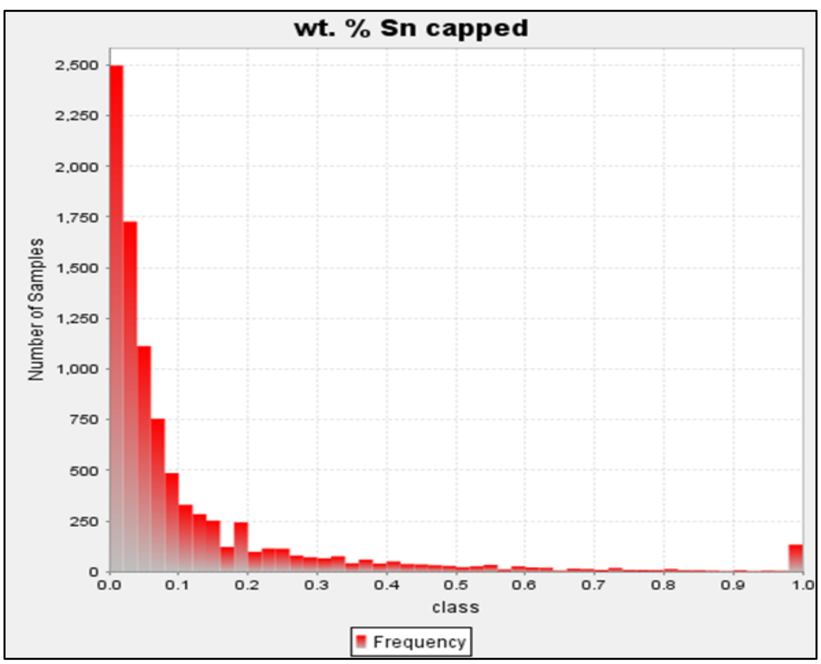
14.8 CAPPING

Following analysis of the Sn assay data distribution observed in the cumulative probability diagram (Figure 14.11) and the identification of a sharp decrease in data density above 1 wt. % Sn, the assays were capped at 1.0 wt.% Sn prior to compositing to remove the high-grade outliers. This approach agrees with the capping limits in previous resource estimates prepared for the East Kemptville deposit (Wardrop, 2010; Avalon Rare Metals Inc., 2014) and avoids the smearing of high grades during compositing. Capping the individual tin assays at 1 wt.% reduces the mean tin grade in the granite geology solid from 0.1419 to 0.1148 wt. %, corresponding to a decrease of 19.1% (Table 14.6). The length-weighted mean was reduced from 0.1173 to 0.1063 wt.% Sn by capping, corresponding to a decrease of 9.4% (Table 14.6).

Table 14.6
Evaluation of the Effect of Capping on the Sn assay Statistics

Statistic Description	All assay data (N = 14,055)		Granite geology solid (N = 9,178)	
	wt. % Sn	wt. % Sn Capped	wt. % Sn	wt. % Sn Capped
Minimum	0.0000	0.0000	0.0000	0.0000
Maximum	27.2000	1.0000	27.2000	1.0000
Mean	0.1084	0.0895	0.1419	0.1148
Standard deviation	0.4886	0.1592	0.5921	0.1775
Length-weighted mean	0.0906	0.0826	0.1173	0.1063

Figure 14.12
Histogram for the Sn Assays in the Granite Geology 3D Solid after Capping at 1 wt.% Sn
(bin size = 0.02 wt.%)



14.9 COMPOSITES

The assay data was composited in 3 m intervals within the granite geology solid from the top down of the drill holes (Figure 14.13). A length of 3 m was chosen because the median sample length is 3.00 m. Very few mineralized sample intervals were found outside of the granite. The last interval in each hole was only created when it was >1.5 m long. Composites that contained <50% sample were deleted. The compositing reduced the average tin grade in the granite geology solid from 0.1148 to 0.1060 wt.% Sn (-7.7%) and the length-weighted mean by 0.3% (Table 14.6).

Figure 14.13
Plan Map Showing the Composites, Drill Hole Traces and the Surface Projection of the Granite

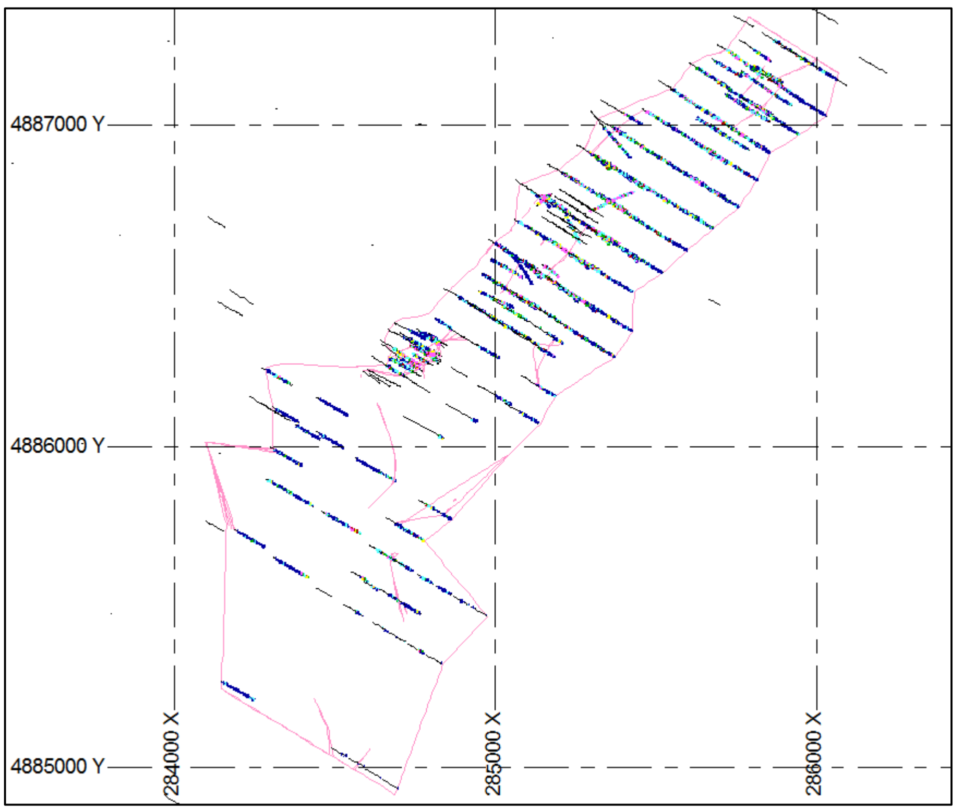


Table 14.7
Evaluation of the Effect of Compositing on the Sn Assay Statistics

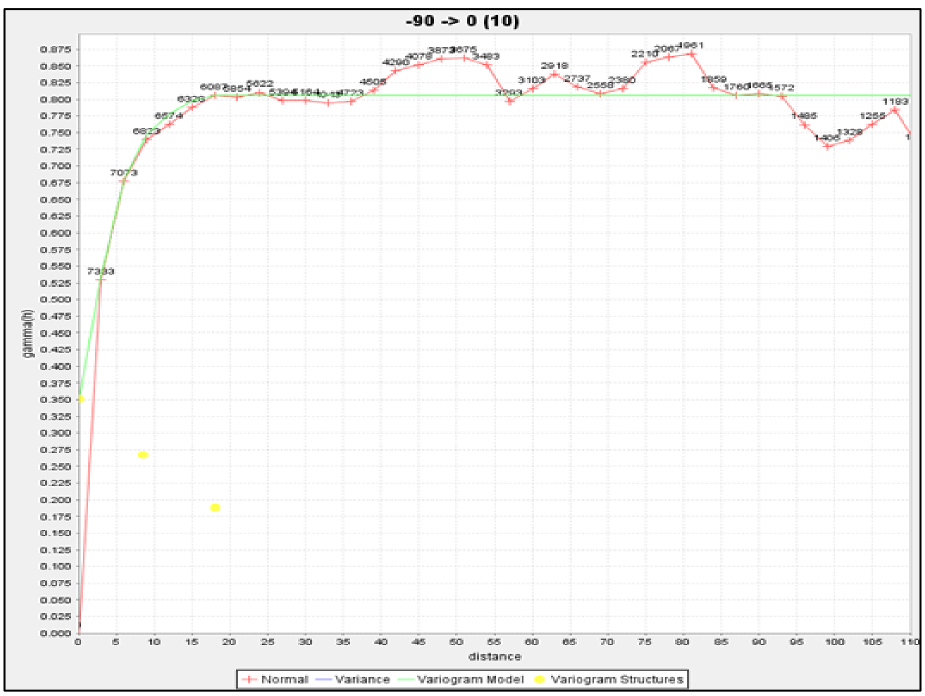
Statistic Description	Granite Solid Assays (N = 9,178)	Granite Solid Composites (N = 7,621)
	wt.% Sn capped	wt.% Sn
Minimum	0.0000	0.0000
Maximum	1.0000	1.0000
Mean	0.1148	0.1060
Standard deviation	0.1775	0.1429
Length-weighted mean	0.1063	0.1060

For the resource estimation, the composites were subdivided spatially into subsets, resulting in a count of 4,277 composites from 193 drill holes for the Main Zone and 762 composites from 24 drill holes for the Baby Zone, allowing some composites to be used for the estimation of both zones. These composites were extracted from a total of 194 drill holes totalling 21,456 m drilled between 1979 and 1991 by previous operators and 23 holes totalling 4,190 m drilled by Avalon in 2014.

14.10 TIN GRADE VARIOGRAPHY

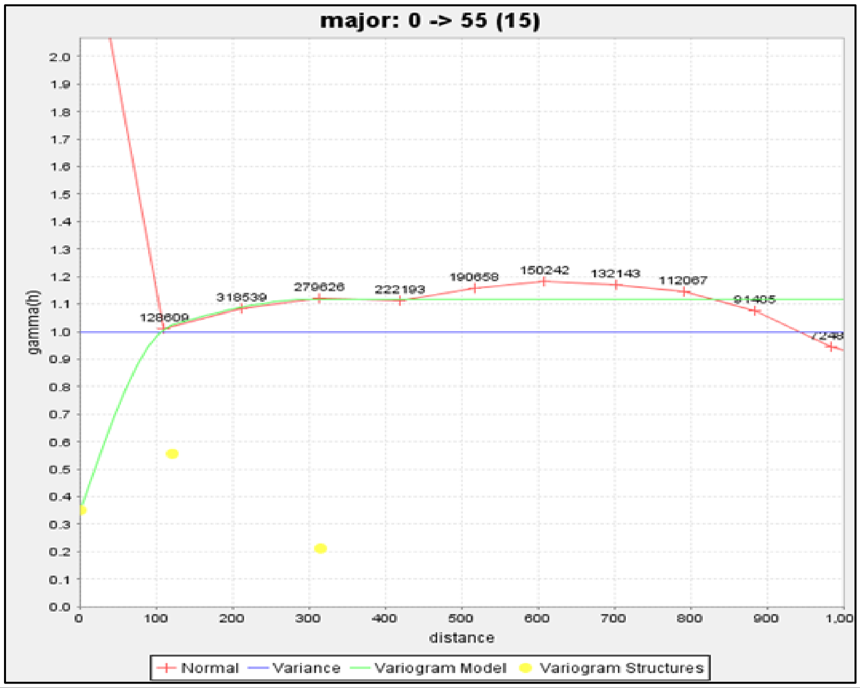
The variography for the tin grades was performed on separate subsets of the tin composites for the Main Zone and the Baby Zone. Separate variograms were modeled because the average sample spacing in the Baby Zone (approximately 22 m) is much closer than that in the Main Zone (approximately 65 m). The downhole semi-variogram for the tin composites in the Main Zone, constructed with a lag distance of 3 m, indicates a range of 18 m and a nugget of 0.35 (Figure 14.14).

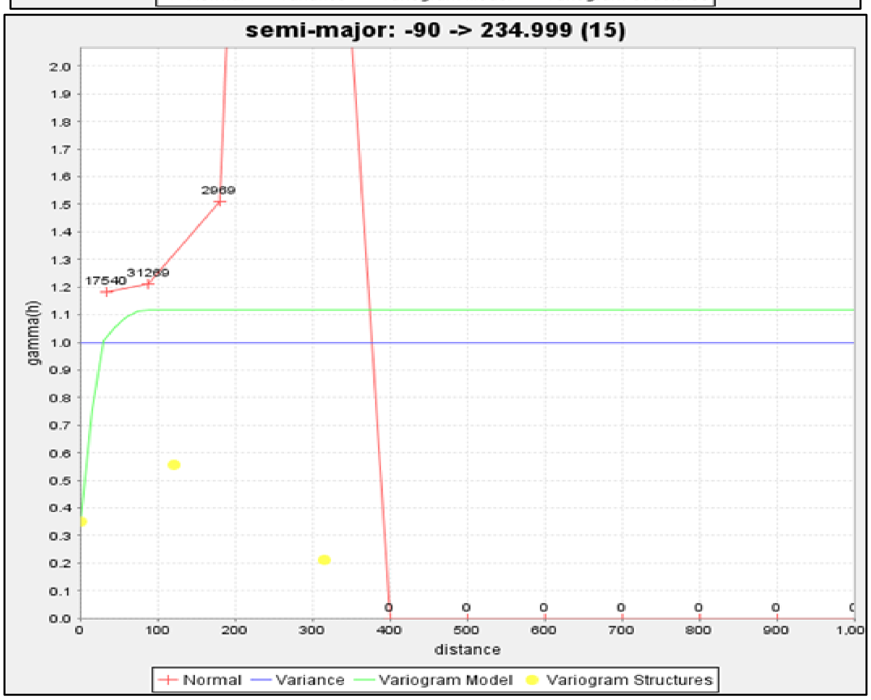
Figure 14.14
Linear Downhole Semi-Variogram for the Tin Composites (Main Zone)

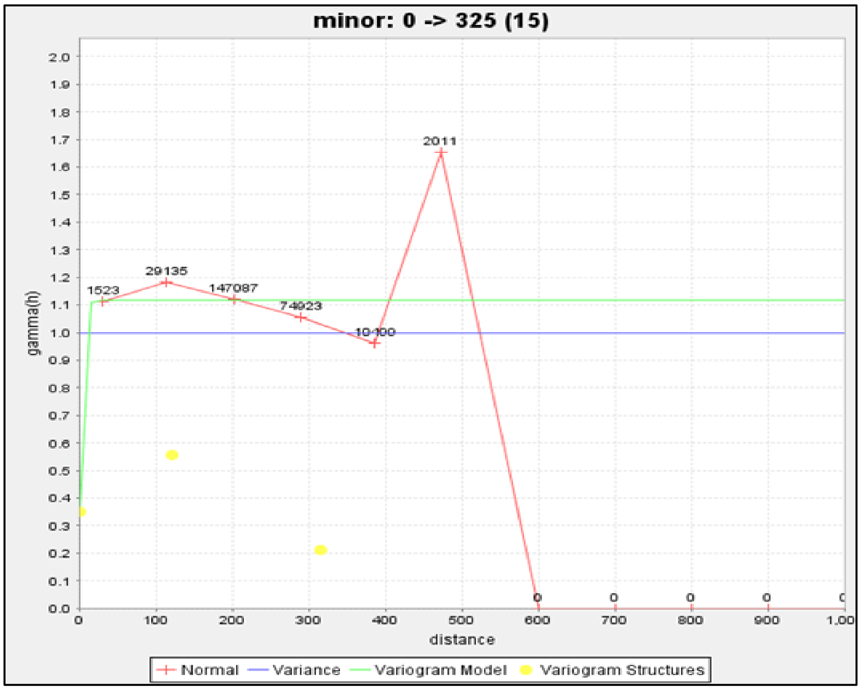


The 3D semi-variograms for the Main Zone were constructed using a lag distance of 100 m and the downhole nugget value was used to model the variograms. The directions of lowest variance, maximum range and highest number of pairs were found to be 55° azimuth / 0° dip with a range of ~315 m (major axis), -90° dip with a range of ~86 m (semi-major axis) and 325° azimuth / 0° dip for which the range of 18 m from the linear downhole semi-variogram was used (minor axis, Figure 14.15). The major (55° azimuth) and semi-major (-90° dip) directions correspond to the strike of the deposit and the generally subvertical dip of the cassiterite greisen veins respectively.

Figure 14.15
Empirical and Modeled Variograms for Sn Composites in the Granite of the Main Zone

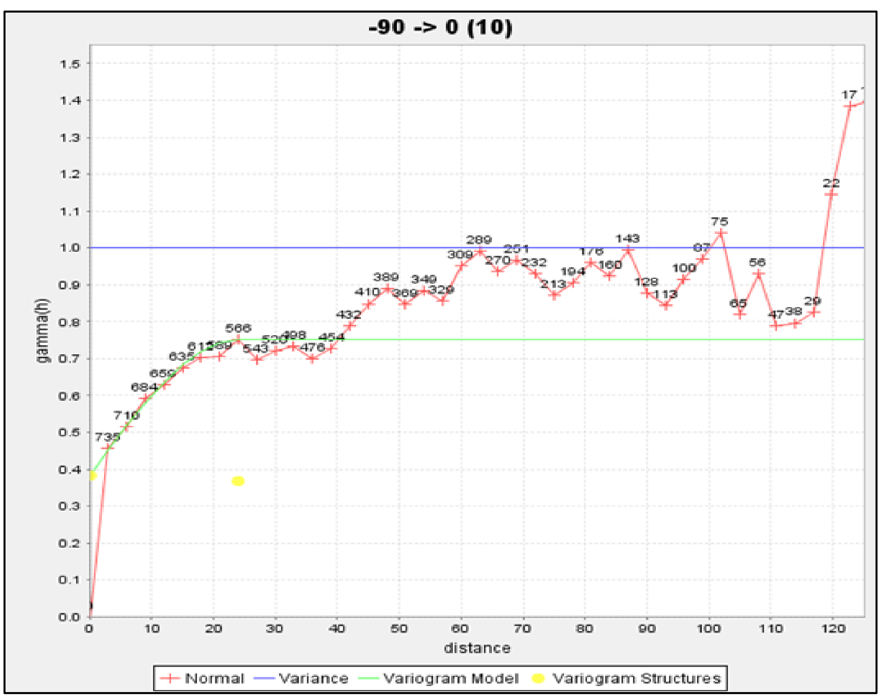






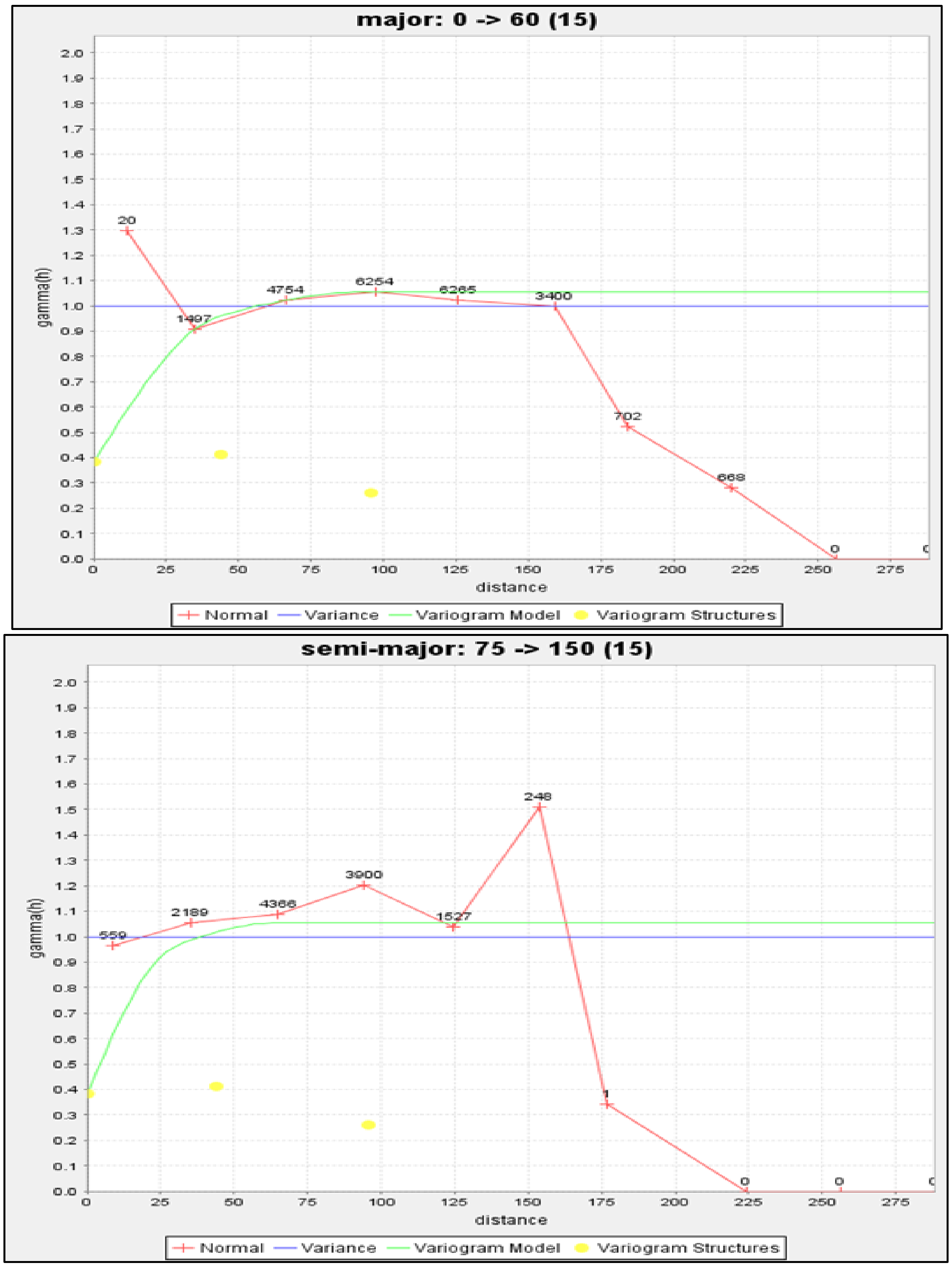
For the Baby Zone, the downhole semi-variogram for the composites, constructed using a lag of 3 m, indicates a range of 24 m and a nugget of 0.38 (Figure 14.16).

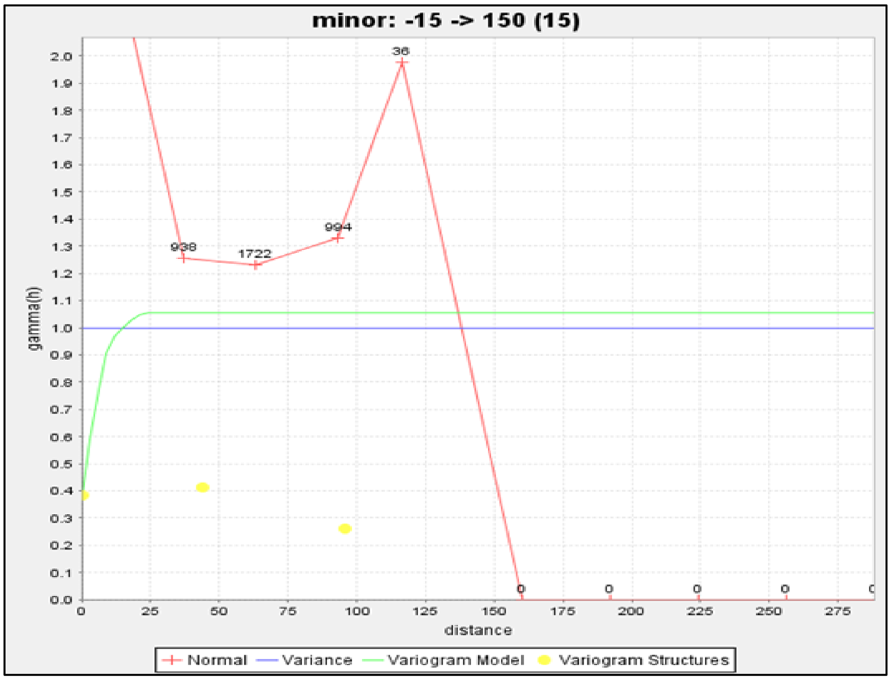
Figure 14.16
Linear Downhole Semi-Variogram for the Tin Composites in the Baby Zone



The 3D variography for the tin composites in the Baby Zone indicates that an ellipsoid with the semi-variograms along 60° azimuth / 0° dip with a range of 95 m (major axis), 150° azimuth / 75° dip with a range of 65 m (semi-major axis) and 150° azimuth / -15° dip with a range of 25 m (minor axis) represent the tin grade continuity of the zone (Figure 14.17). The variograms were calculated with a lag of 35 m and the downhole nugget of 0.38 was used to model the variograms.

Figure 14.17
Empirical and Modeled Variograms for Sn Composites in the Granite of the Baby Zone





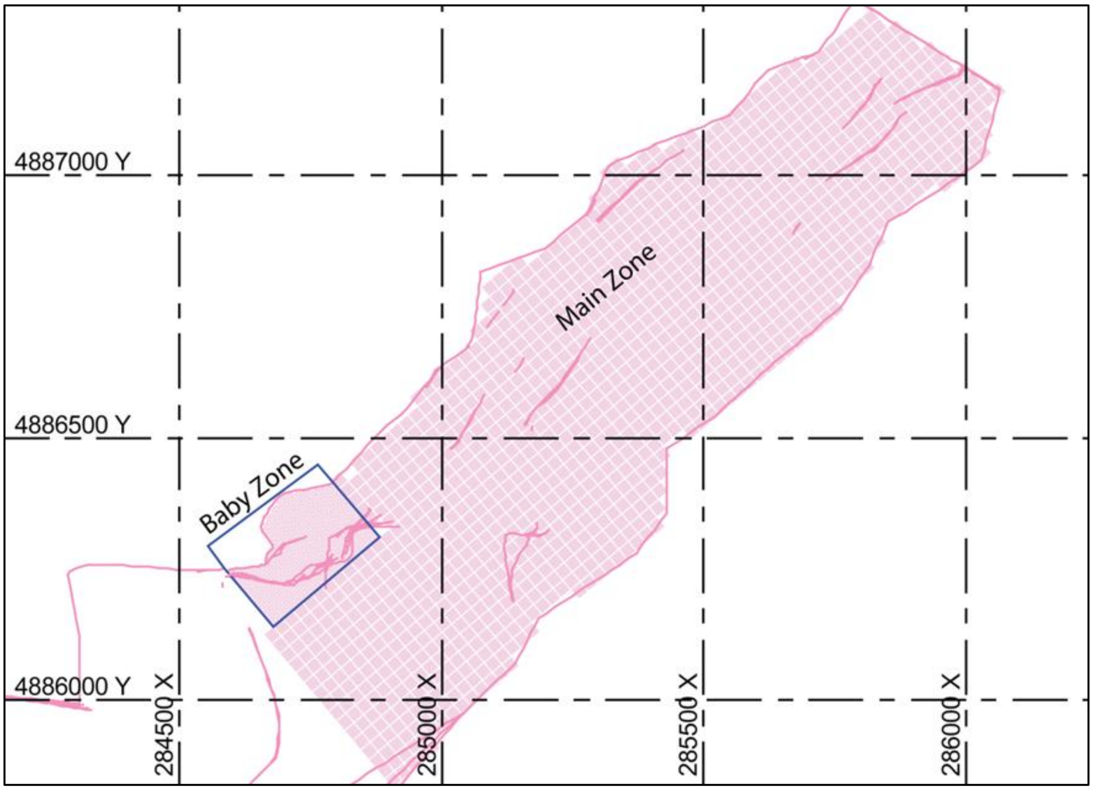
14.11 BLOCK MODEL AND GRADE ESTIMATION

Two separate block models were developed which accommodate the wide drill spacing of the Main Zone and the greater detail of the drilling data for the Baby Zone (Figure 14.18). Following the interpolations, the block models were re-blocked and merged into one 6 x 6 x 6 m model and provided to Micon for the purpose of developing of a combined mine design and schedule. The resources are reported for the two separate block models for the Main and Baby Zones.

14.11.1 Block Model Dimensions

For the Main Zone, a block size of 24 m x 24 m width and 12 m height was chosen to accommodate the drill section spacing of ~100 m and the bench height of the historic mining (12 m; Table 14.8). For the Baby Zone, a small block size of 6 m in all dimensions was chosen which allows a high level of detail for the grade interpolation and rock code assignment and is amenable to subsequent merging with the Main Zone block model. Both block models were rotated 40° from east-west in GEMS (corresponding to a row alignment along a 50° azimuth) approximately parallel to the strike direction of the granite. The origins were chosen so that they coincide approximately with the existing bench levels and so that the block models can be merged.

Figure 14.18
Plan of the two Main and Baby Zones Block Models



Note: Only blocks with the granite rock code are shown, the outline of the granite wireframe projected to surface is shown for reference. The thin red lines are projections of the contacts.

Table 14.8
Main Zone and Baby Zone Block Model Dimensions

Description	Main Zone	Baby Zone
Block size (m)		
Column	24	6
Row	24	6
Level	12	6
Origin (m)		
X	285,000	284,689.33
Y	4,884,000	4,885,901.07
Z	154	124
Rotation angle	40°	40°
Number of blocks		
Column	150	100
Row	125	100
Level	50	50

14.11.2 Block Model Parameters

The parameters, range of values and the methods for their assignment for both block models are summarized in Table 14.9.

Table 14.9
Block Model Parameters

Parameter	Type[1]	Description	Values
Rock Type	Integer	Assigned from 3D geology solids and topography based on a 50% threshold using needling level 10 (100 needles per block) vertically down levels.	0 = air 1 = Unknown (outside of geology models) 2 = Metasediment 3 = Granite 5 = Overburden
Density	Single		Main Zone granite = 2.728 Baby Zone granite = 2.784 Metasediment = 2.756 Unknown = 2.756 Overburden = 1.8
SNPCT	Single	tin grade in percent, assigned by Kriging	
Zone	String	based on geology and drilling coverage	Main-NE = northeast granite Baby = Baby Zone
Mined	Integer	mined out, = blocks with rock code 2 & 3 50% above the 1992 topography, using 10 needles vertically down	0 = not mined out 1 = mined out
% air	Double	percentage of block within air solid	
% Granite	Double	percentage of block within granite geology solid	
% Metased	Double	percentage of block in metasediment geology wireframes	
% Overburden	Double	percentage of block within overburden solid	
Pass	Integer	interpolation pass	1 = first interpolation pass 2 = second pass 3 = third pass
Blockvar	Double	block variance	
Gr-cl-pt	Double	grade of closest point to block	
Nearest	Double	actual distance to closest point to block	
No-of-holes	Integer	number of holes used to interpolate block	
No-of-points	Integer	number of points used to interpolate block	
MII	Integer	resource category	1 = measured 2 = indicated 3 = inferred
Krig-Var	Double	Kriging variance	

[1]Integer = whole number without decimals, Single, Double = high-precision numbers with decimals.

14.12 TIN GRADE ESTIMATION METHODOLOGY

The tin grade of all blocks containing >0% granite was interpolated using Ordinary Kriging in the Geovia GEMS 6.8.1 software (Dassault Systems). Separate search ellipses and variograms based on the variography were used for the Main Zone and Baby Zone. The tin grade was interpolated in three subsequent passes (1, 2, 3), corresponding approximately to the measured, indicated and inferred categories. For each pass and zone, the search ellipse parameters are listed in Table 14.10. The search ellipse dimensions were adjusted proportionally from the values determined by variography.

Trial interpolations were performed with more limited search ellipse radii for grades >0.5 wt. % Sn, but contrary to the present method, these did not successfully reproduce the known grade and tonnage of the mined material (see Section 14.14.6). The number of composites used to interpolate each block was limited to six per drill hole in the Main Zone and to two per drill hole in the Baby Zone. Combined with the restrictions for the number of samples (Table 14.10, Table 14.11), this results in 3 to 6 holes used for the estimation of the measured and indicated category blocks and 2 to 4 holes used for the estimation of inferred blocks. For the discretization during interpolation, the blocks were subdivided into three in the X- and Y-directions and into two in the Z-direction for the Main Zone and into two in each dimension for the Baby Zone.

In the Main Zone, a constrained maximum range of 40 m was chosen for the measured category (Pass 1). For the indicated category (Pass 2), a maximum range of 120 m was selected to allow interpolation across the 100 m-spaced drill sections. For Pass 3 (inferred category), the search ellipse determined by variography was used in order to interpolate the grades of blocks in the granite wireframe. The effect of limiting the radius of the major axis of the Pass 3 search ellipse to 160 m instead of 315 m was evaluated and yielded only a small reduction in tonnage of the inferred category (-0.91 Mt at a cut-off grade of 0.100 wt. % Sn and -0.66 Mt at a cut-off grade of 0.12 wt. % Sn) and insignificant overall grade changes.

In the Baby Zone, the maximum ranges for the measured (Pass 1) and indicated search ellipses (Pass 2) correspond to approximately 1/3 and 1/2 of the range of the major axis determined by variography. The inferred (Pass 3) major axis range was set to that determined by variography.

Table 14.10
Interpolation Parameters for the Main Zone

Pass	Resource category	Search ellipse			No. of Samples
		X	Y	Z	
1	Measured	40	20	15	18-36
2	Indicated	120	40	18	18-36
3	Inferred	315	85	18	12-24

Table 14.11
Interpolation Parameters for the Baby Zone

Pass	Resource category	Search ellipse			No. of Samples
		X	Y	Z	
1	Measured	30	20	8	6-12
2	Indicated	48	33	12	6-12
3	Inferred	95	65	24	4-8

14.13 RESOURCE CLASSIFICATION

The East Kemptville mineral resource was classified based on the Passes from the Kriging interpolation (Table 14.10, Table 14.11 above) which correspond to search ellipses, in combination with the requirement that measured and indicated resources are characterized by geological continuity that is well-constrained by drilling in three dimensions. At least three drill holes were used to estimate measured and indicated blocks. In some cases, Pass 2 or 3 blocks were included in zones classified as measured or indicated where they were surrounded by Pass 1 or 2 blocks, geological continuity was indicated and sufficiently spaced drill holes were present. In addition to geological observations and the variography, the rationale for classifying blocks as indicated resources in the Main Zone using a search ellipse measuring 120 x 40 x 18 m is that the previous mining has demonstrated that the deposit is continuous along strike at a ~50° azimuth across the ~100 m drill sections.

14.14 BLOCK MODEL VALIDATION

The block model was validated in cross-sections and plan maps, by comparing the block model volumes to the wireframes, and by comparing the block model grades to the assays, composites and other methods of grade interpolation.

14.14.1 Validation of the Volume Assignment

A comparison of the pre-mining volume of the granite 3D wireframe (134,364,183 m^3) to the blocks assigned >50% granite percentage (19,326 blocks ≙ 133,581,312 m^3) in the 24 x 24 x 12 m block model that covers the Main Zone shows that the blocking resulted in a -0.6% volume loss. The rock percentage, forming the basis of the tonnage calculation, was thus correctly assigned from the 3D wireframe.

14.14.2 Validation in Cross-Sections and Level Plans

For the Main Zone, representative level plans (Figure 14.19 to Figure 14.21) and cross-sections (Figure 14.22 to Figure 14.27) were prepared which show the results of the Kriging grade interpolation and the grade of the drill hole composites. The effect of the shape of the search ellipse is clearly visible in the plan views (Figure 14.19 to Figure 14.21). The cross-sections indicate that the interpolated grades are consistent with those of the composites (Figure 14.22 to Figure 14.27).

Figure 14.19 details the Main Zone, Level 6 plan view showing tin grade in wt. % of in situ blocks and drill hole composites. The outline of the historic pit is visible a white area of mined out blocks.

Figure 14.19
Main Zone, Level 6 Plan View Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites (The outline of the historic pit is visible a white area of mined out blocks) Zones Block Models

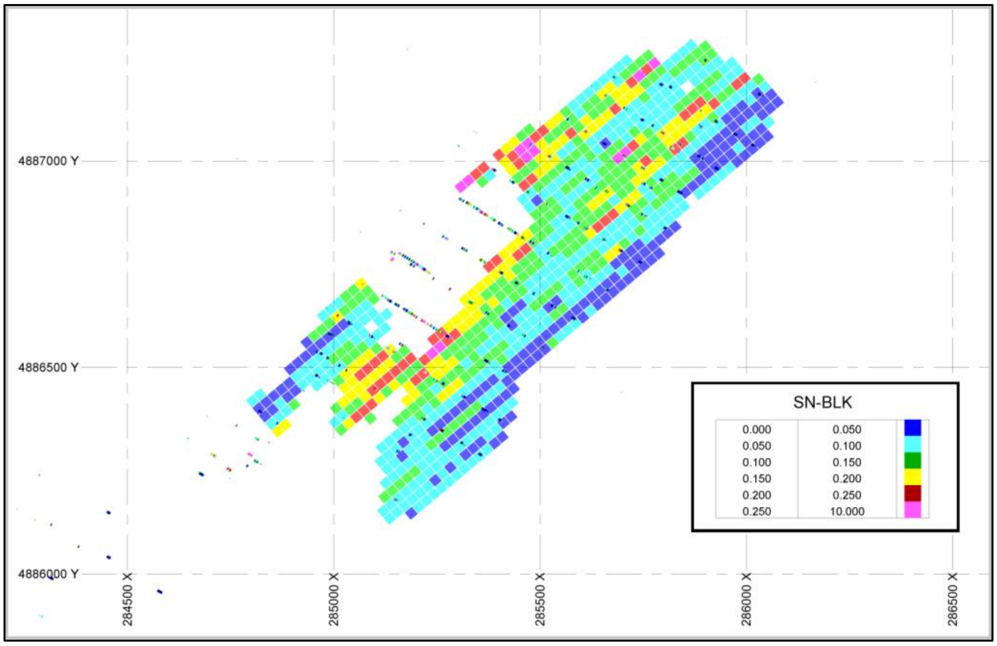


Figure 14.20
Main Zone, Level 8 Plan View Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites

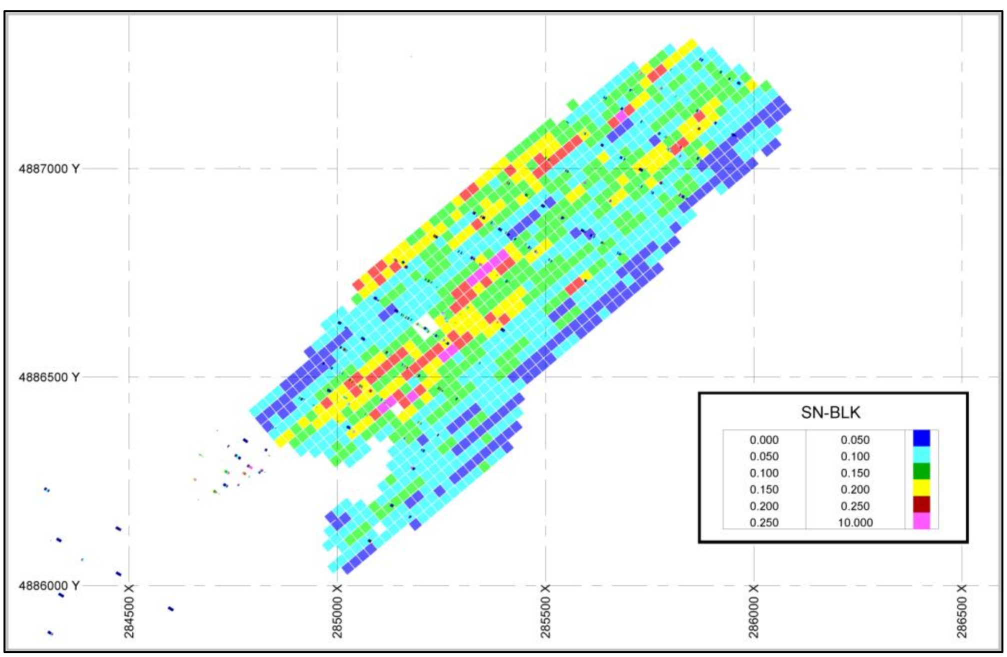


Figure 14.21
Main Zone, Level 10 Plan View Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites

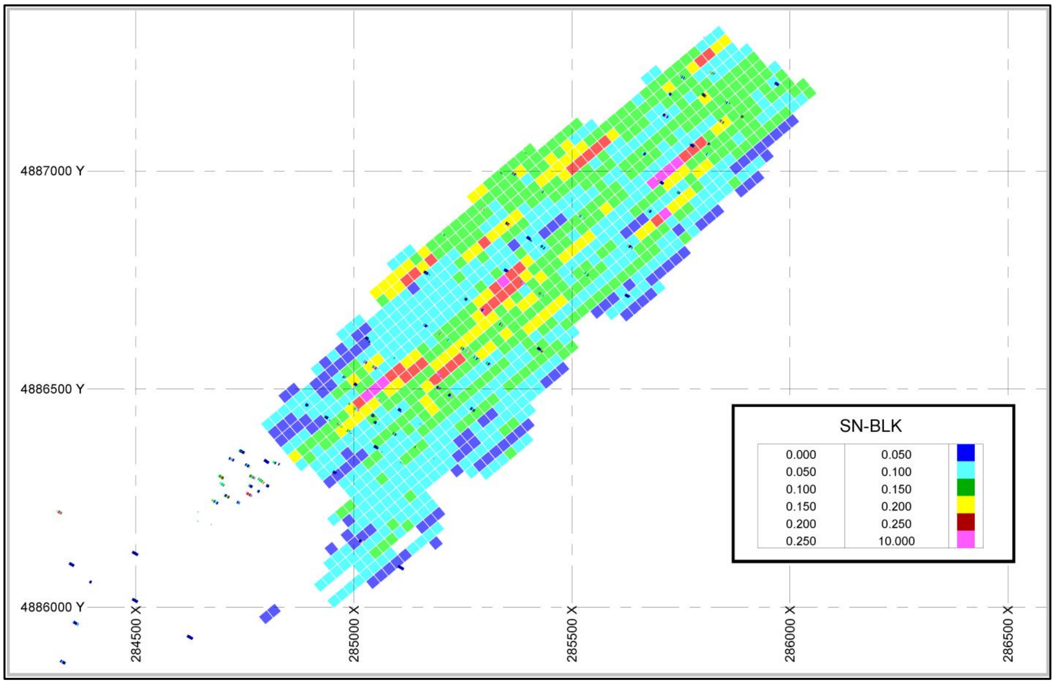


Figure 14.22
Main Zone, Column 67 Section Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites
(granite outline – pink, topography –brown)

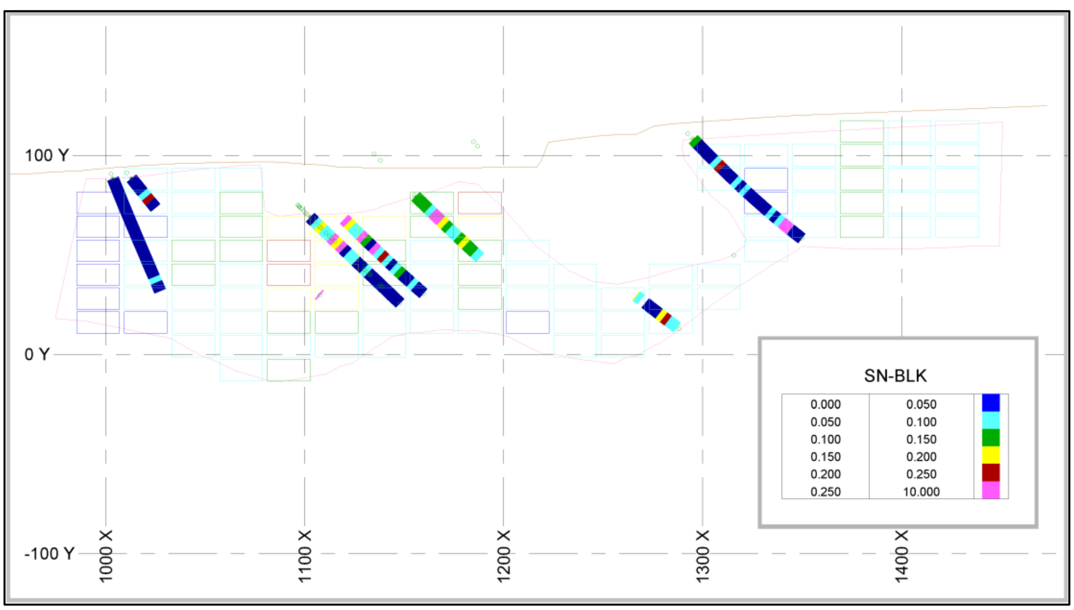


Figure 14.23
Main Zone, Column 74 Section Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites (granite outline – pink, topography –brown)

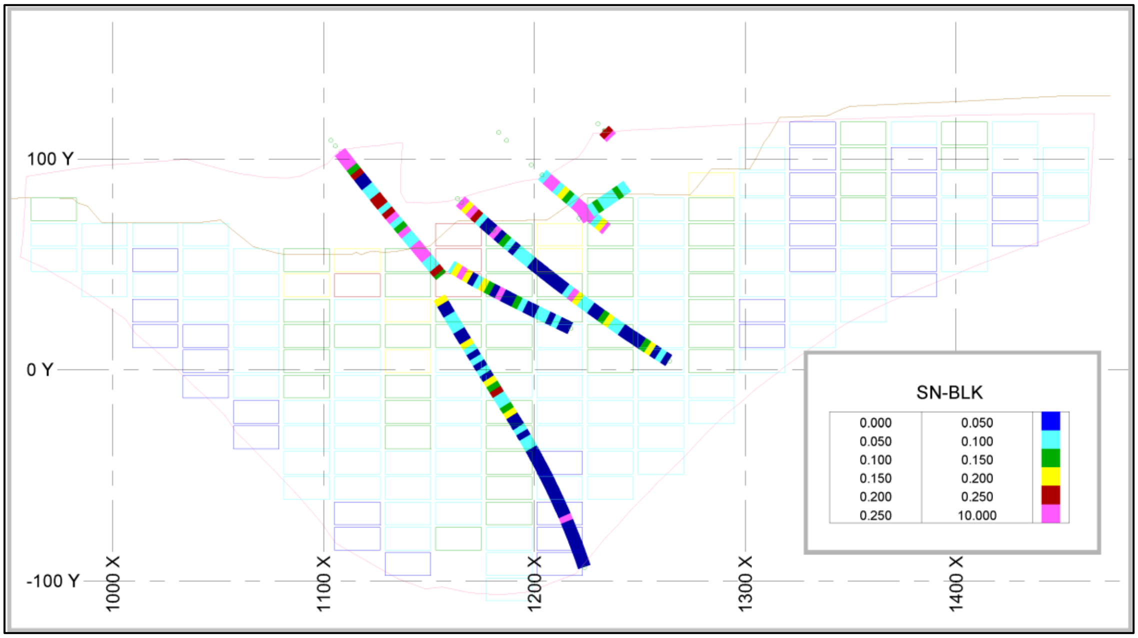


Figure 14.24
Main Zone, Column 92 Section Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites (granite outline – pink, topography –brown)

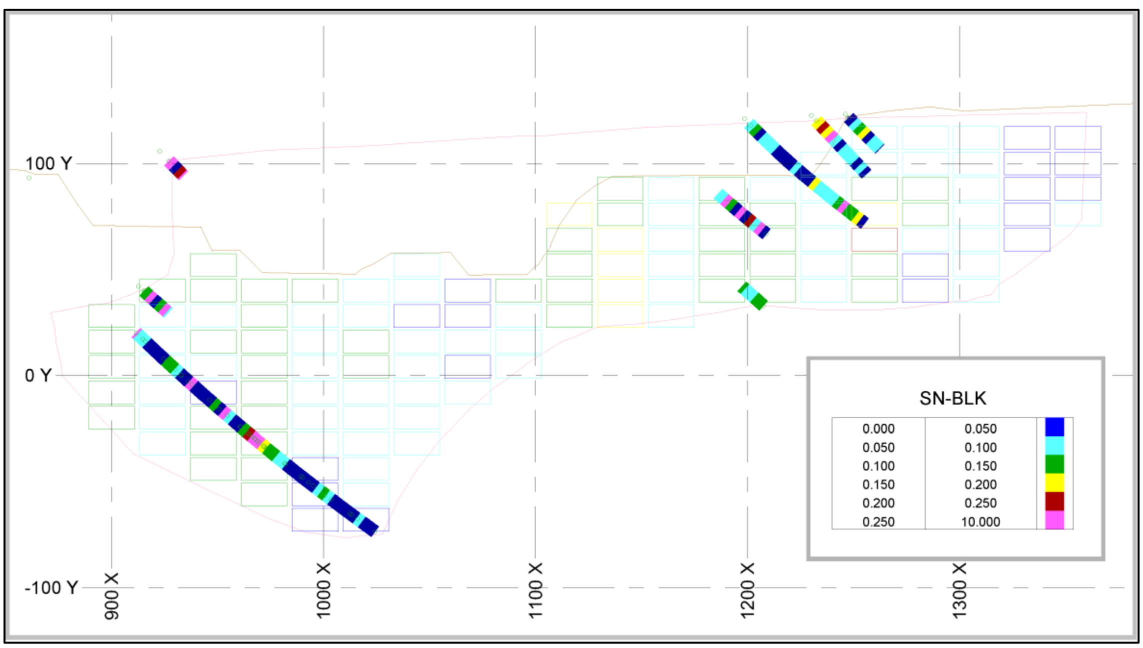


Figure 14.25
Main Zone, Row 43 Section Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites (granite outline – pink, topography –brown)

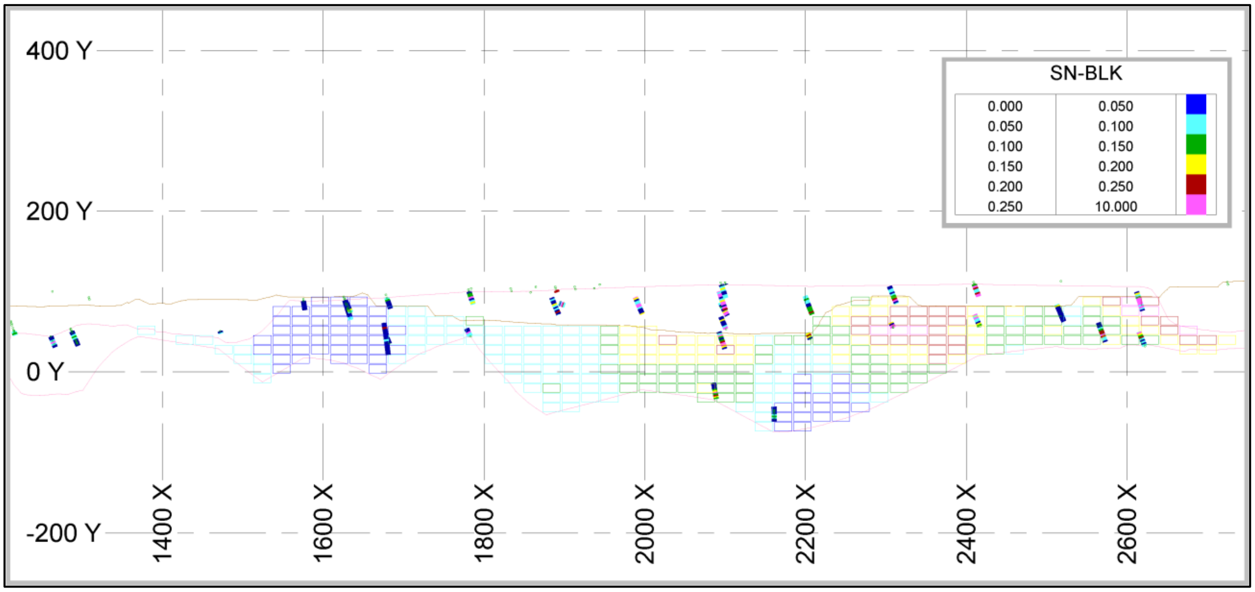


Figure 14.26
Main Zone, Row 47 Section Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites (granite outline – pink, topography –brown)

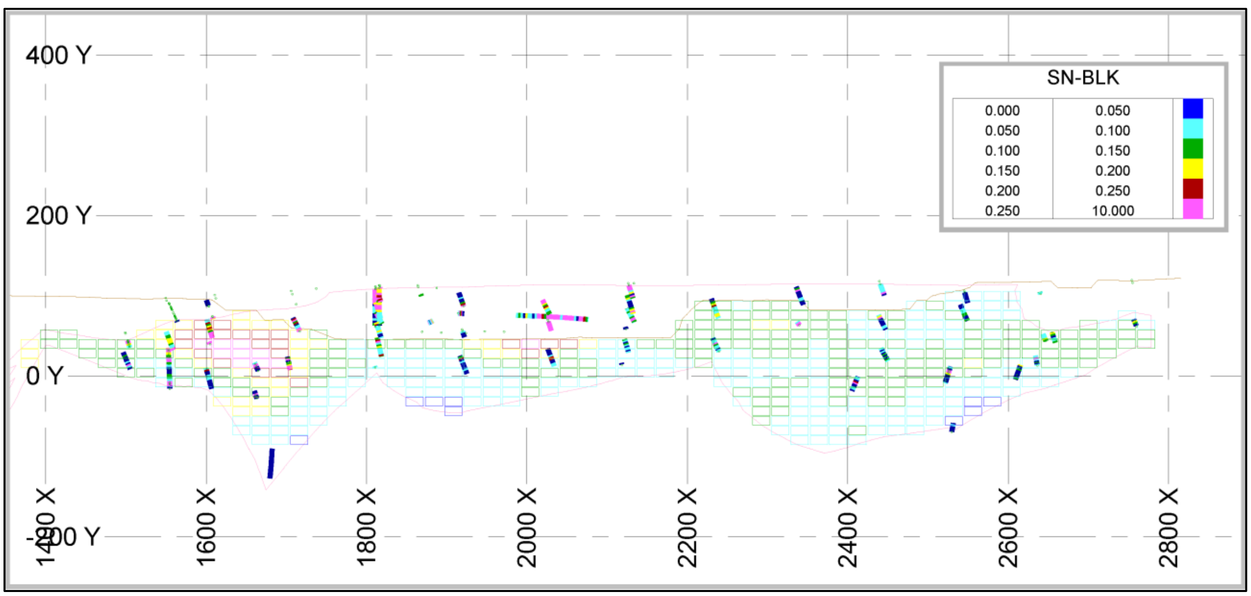
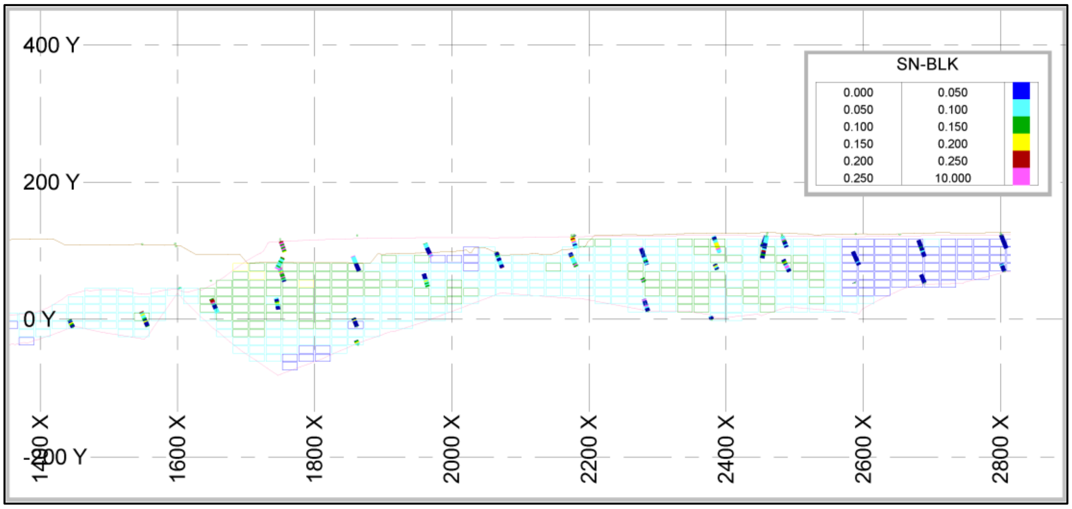


Figure 14.27
Main Zone Row 53 Section Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites
(granite outline – pink, topography –brown)



For the Baby Zone, the representative level plans in Figure 14.28 and Figure 14.29 and the cross-sections (Figure 14.30 and Figure 14.31) also indicate an acceptable grade interpolation that is consistent with the composite data.

Figure 14.28
Baby Zone, Level 10 Plan Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites
(granite outline – pink, topography –brown)

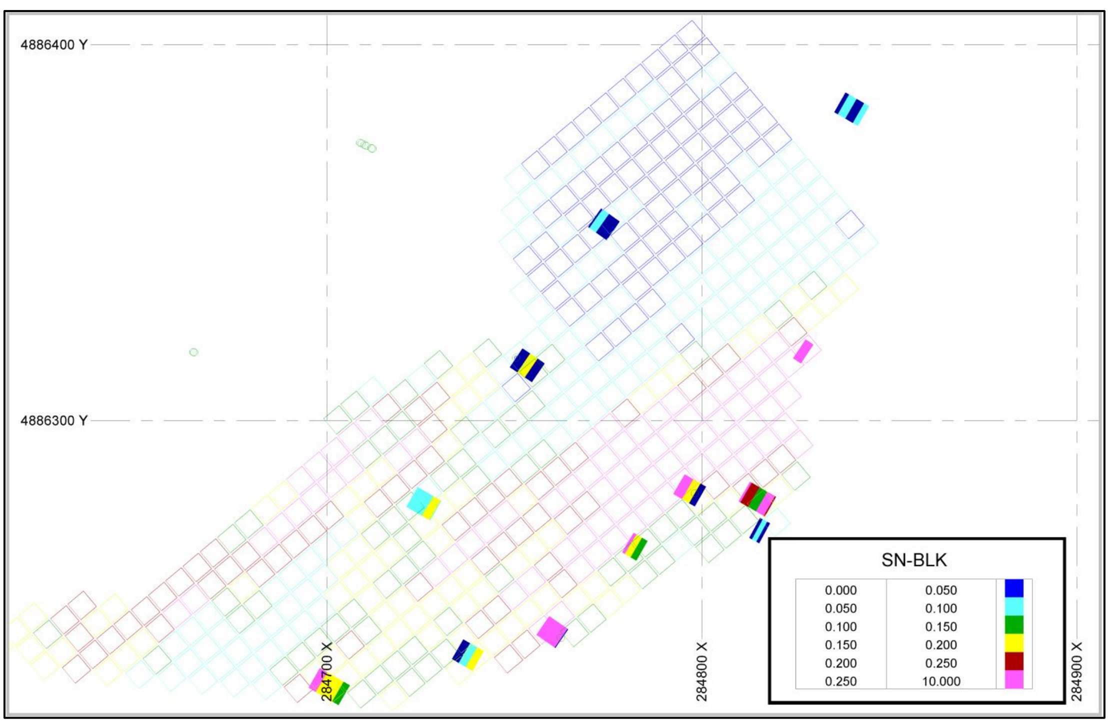


Figure 14.29
Baby Zone, Level 18 Plan Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites
(granite outline – pink, topography –brown)

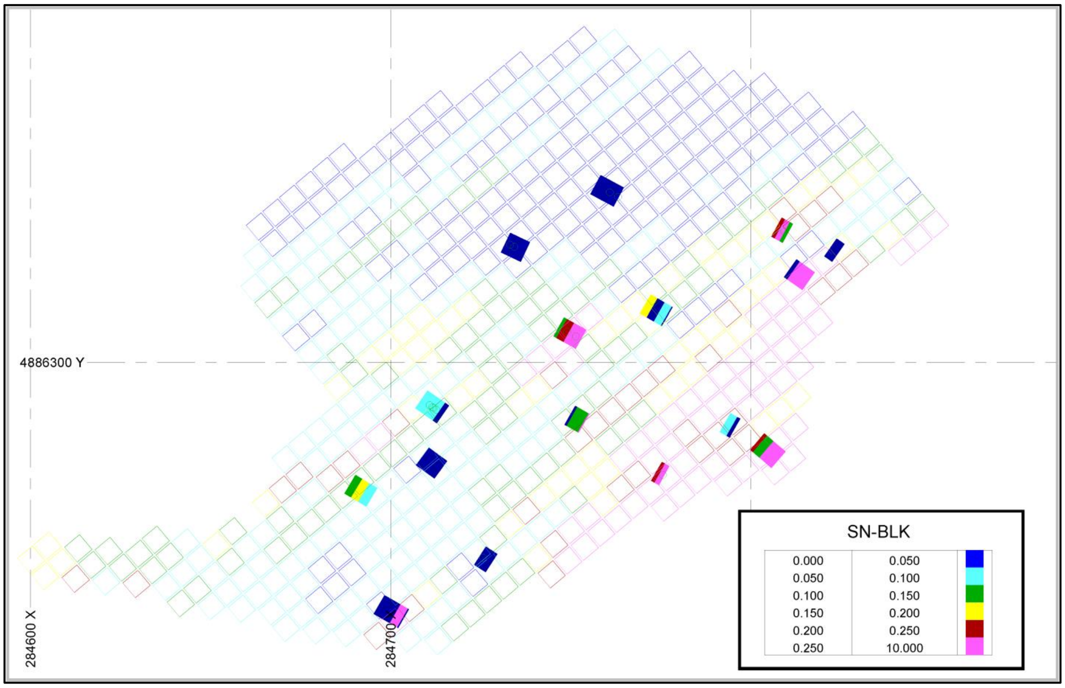


Figure 14.30
Baby Zone, Row 59 Section Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites
(granite outline – pink, topography –brown)

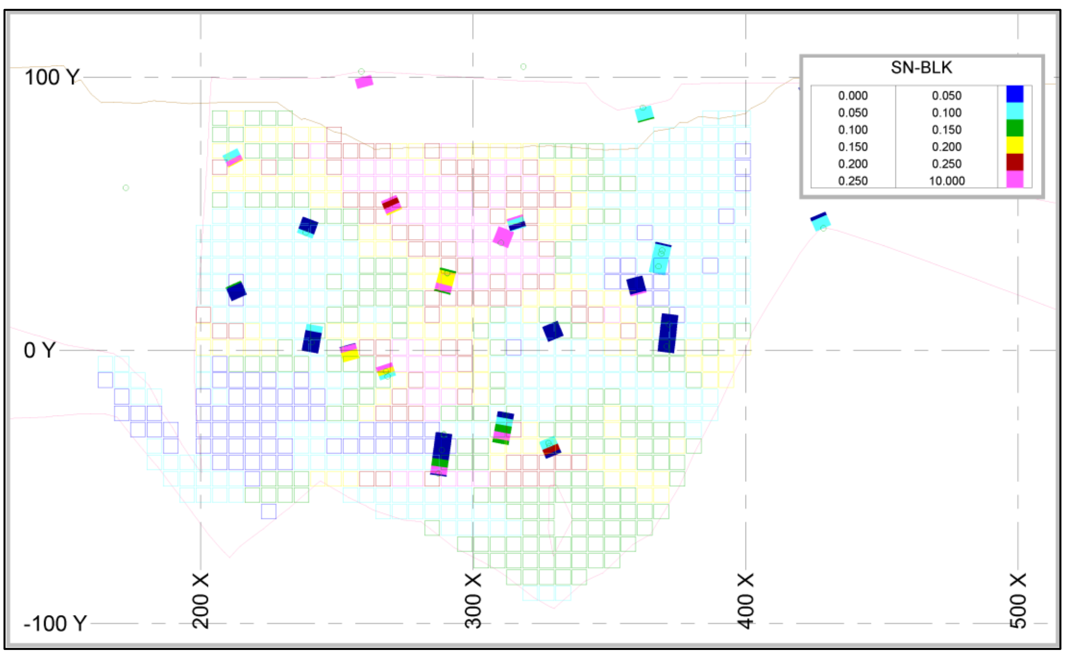
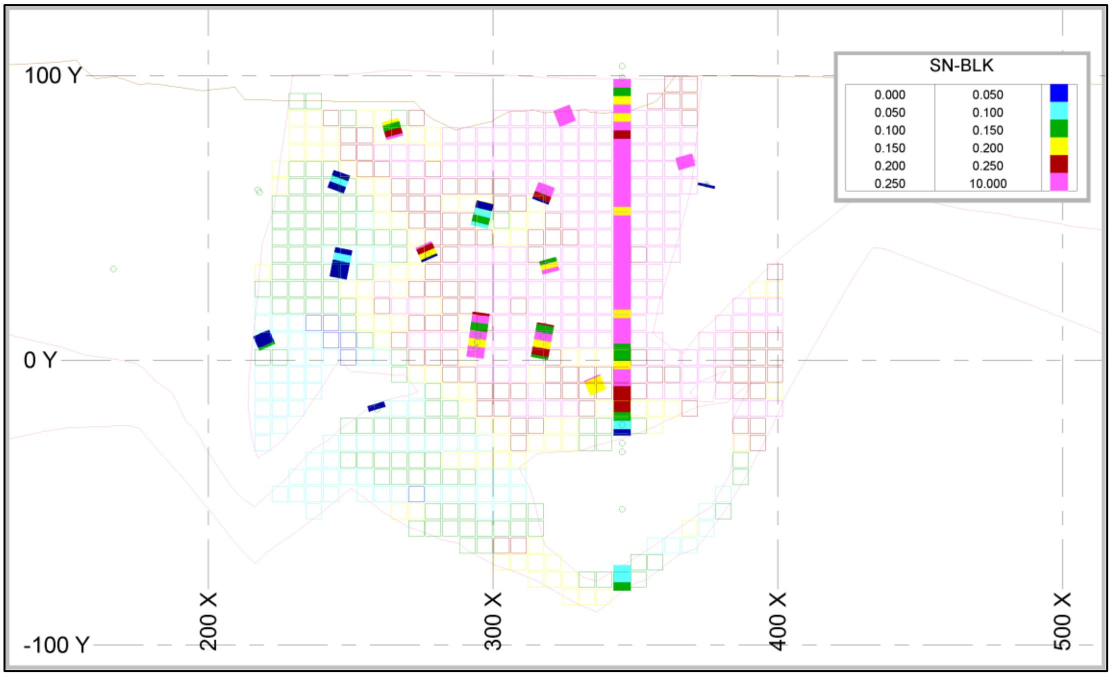


Figure 14.31
Baby Zone, Row 62 Section Showing Wt.% Sn of In-situ Blocks and Drill Hole Composites
(granite outline – pink, topography –brown)



14.14.3 Comparison to the Composite Data

The mean grades of all interpolated blocks of the Main and Baby Zones were compared to the mean grades of the corresponding composites (Table 14.12). For the Main Zone, there is no difference between the mean grade of the composites and that of the blocks, whereas the mean grade of the blocks is lower than that of the composites in the Baby Zone.

The GEMS software provides a function to calculate the spatially de-clustered mean of the composites, which was performed using cell sizes of 24 x 24 x 12 m and 6 x 6 x 6 m for the Main and Baby Zones, respectively (Table 14.12). Although the de-clustering does not have a large effect on the average tin grade of the Baby Zone, inspection of the locations in 3D shows that the higher mean of the composites is the result of the close spatial proximity of the data. For the Main Zone, de-clustering of the composites yields the same result as the raw composites. This analysis indicates that the grades were not overestimated using the Kriging interpolation.

Table 14.12
Comparison Between the Mean Block Model and Composites Tin Grades

Zone	Block model Sn (wt. %)	Composites Sn (wt. %)	De-clustered composites Sn (wt. %)
Main Zone	0.093 (N = 7582)	0.093 (N = 4277)	0.093 (N = 4277)
Baby Zone	0.116 (N = 12539)	0.145 (N = 762)	0.140 (N = 762)

14.14.4 Comparison between Kriging and GEMS "Assign Grade from Drill Holes" Function

The GEMS software provides a function to assign an average grade to 3D wireframes directly from the composites of intersecting drill holes. For this purpose, 3D outlines around all resource categories for the Main Zone and the Baby Zone were generated and the grade was calculated from the same drill hole composites that were used for the resource estimate. This methodology can be considered to be producing a very simple resource model that may be utilized for comparison and verification purposes. The results, presented in Table 14.13, are similar to the comparison with the average grade of the composites (Table 14.12), and show that the Kriging interpolation produces lower mean grades for the overall volumes than the averages of the drill holes, particularly for the Baby Zone, but overall confirms the general validity of the grades.

Table 14.13
Comparison Between the Tin Grades (in wt. % Sn) of the Block Model and Those Assigned to the 3D Outline of All Resource Categories

Volume	Block Model Grade (Wt.% Sn)	Grade from Drill Holes (Wt.% Sn)
Main Zone	0.093 (N = 7582)	0.100
Baby Zone	0.1160 (N = 12539)	0.150

14.14.5 Comparison Between Kriging and Inverse Distance Weighting Tin Grade Interpolation

The tin grade interpolation using Ordinary Kriging was validated by interpolating the same blocks with the same search ellipses but using the Inverse Distance Weighting Squared (IDW^2) method. For the Main Zone, the IDW^2 method yields similar to higher tin grades for all resource categories and cut-off grades, ranging from 0% to 12% difference (Table 14.14). The difference is largest for the measured category and for a higher cut-off grade (Table 14.14). This suggests that the grades calculated using Ordinary Kriging (OK) represent a slightly conservative estimate.

Table 14.14
Comparison Between the Tin Grades Interpolated Using IDW^2 and OK in the Main Zone at Variable Cut-Off Grades
(The tonnages of the two methods are identical as they compare the same blocks)

Category	Cut-off grade											
	none			0.05 wt. % Sn			0.10 wt. % Sn			0.15 wt. % Sn		
	OK	IDW^2	Δ	OK	IDW^2	Δ	OK	IDW^2	Δ	OK	IDW^2	Δ
Measured	0.146	0.157	8%	0.154	0.167	8%	0.177	0.185	5%	0.200	0.223	12%
Indicated	0.109	0.11	1%	0.115	0.119	3%	0.145	0.153	6%	0.189	0.196	4%
Inferred	0.084	0.084	0%	0.096	0.102	6%	0.132	0.142	8%	0.186	0.2	8%

For the Baby Zone, the IDW^2 interpolation yields lower tin grades than those interpolated by Ordinary Kriging for the measured category blocks calculated at cut-off grades from 0 to 0.15 wt. % Sn (-8 to -2%, Table 14.15). This is likely the result of an increased range of influence of higher-grade composites in the Ordinary Kriging method. For the indicated and inferred category blocks, the difference is positive and smaller, ranging from 2 to 9% (Table 14.15).

Table 14.15
Comparison Between the Tin Grades Interpolated Using IDW^2 and OK in the Baby Zone at Variable Cut-Off Grades
(The tonnages of the two methods are identical as they compare the same blocks)

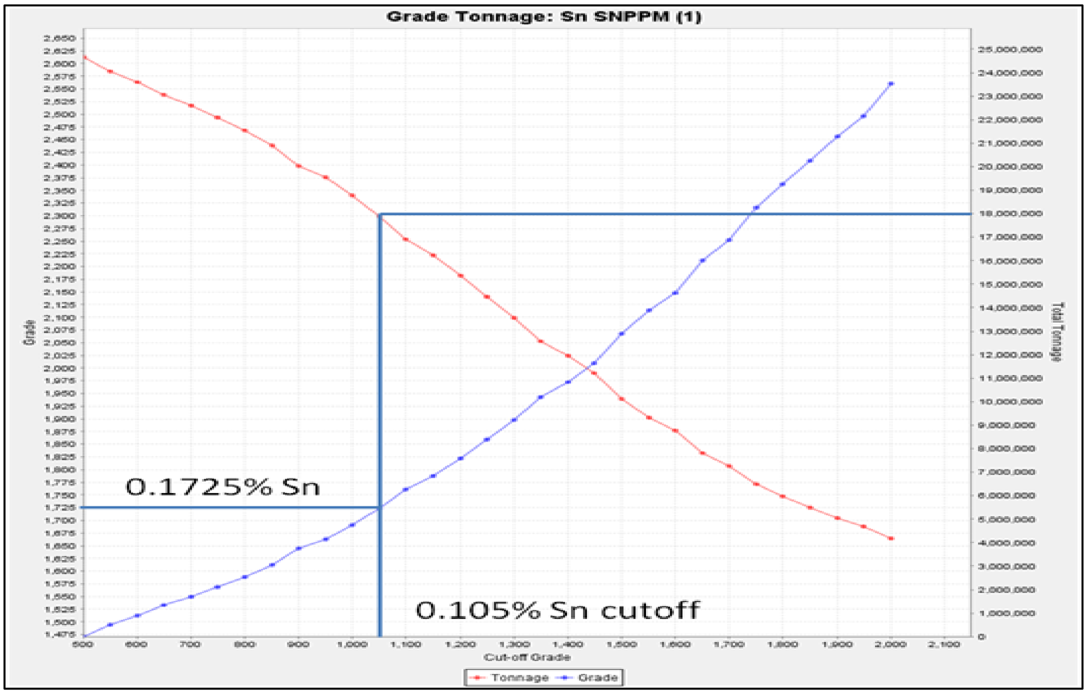
	Cut-off grade											
	none			0.05 wt. % Sn			0.10 wt. % Sn			0.15 wt. % Sn		
Category	**OK**	**IDW^2**	**Δ**	**OK**	**IDW^2**	**Δ**	**OK**	**IDW^2**	**Δ**	**OK**	**IDW^2**	**Δ**
Measured	0.236	0.218	-8%	0.236	0.222	-6%	0.251	0.246	-2%	0.277	0.284	3%
Indicated	0.145	0.148	2%	0.163	0.171	5%	0.207	0.223	8%	0.248	0.271	9%
Inferred	0.091	0.093	2%	0.116	0.124	7%	0.160	0.176	10%	0.206	0.215	4%

Overall, the largest portion of the blocks interpolated using Ordinary Kriging shows lower grades when interpolated via IDW^2, indicating that the Kriging method, which was used for the resource estimation, is more conservative in most cases.

14.14.6 Comparison of the Interpolated Grade to the Mined Material

Historic reports show that the mine production from the Main Zone at East Kemptville was ~18 Mt with a mill head grade of 0.184 wt. % Sn; an operating cut-off grade of ~0.1 wt. % Sn was used. To validate the tin grade interpolation of the current block model, the mined blocks in the Main Zone were interpolated using the same Kriging parameters as in the current resource estimate. This interpolation yields a tonnage that is highly dependent on the cut-off grade (Figure 14.32). At a cut-off grade of 0.105 wt. % Sn, the tonnage of 18 Mt is reproduced, and the overall grade is 0.1725 wt. % Sn (Figure 14.32). The indicates that the current interpolation parameters may underreport the tin grade by 0.012 wt. % or ~6% in the Main Zone and thus provides a conservative tin grade estimate with an acceptable deviation from that of the mined material.

Figure 14.32
Grade-Tonnage Curve for the Interpolation of Mined Blocks in the Main Zone



14.14.7 Grade-Tonnage Curves and Cut-Off Grade

For the Main and Baby Zones, grade-tonnage curves for cut-off grades ranging between 0.02 and 0.20 wt. % Sn in increments of 0.005 wt. % Sn were prepared for all resource categories and the Measured + Indicated categories (Figure 14.33 to Figure 14.36). The plots show that the resource tonnage is very sensitive to the cut-off grade being applied in both zones. For example, in the Main Zone overall resource (all categories) a cut-off grade of 0.100 wt. % Sn results in a tonnage of 54.21 Mt grading 0.138 wt. % Sn and a cut-off grade of 0.110 wt. % corresponds to a tonnage of 42.65 Mt grading 0.147 wt. % Sn (Figure 14.33). The same example for only the measured and indicated resource categories in the Main Zone shows that an increase in cut-off grade from 0.100 to 0.110 would be associated with a tonnage decrease from 26.12 to 21.74 Mt (Figure 14.34).

This implies that very careful grade control will be needed in the mining operation and also has implications for comparing the present estimate to previous resource estimates which used different interpolation methods and different cut-off grades (see Section 14.16).

Figure 14.33
Grade-Tonnage Curve for Blocks in All Resource Categories in the Main Zone

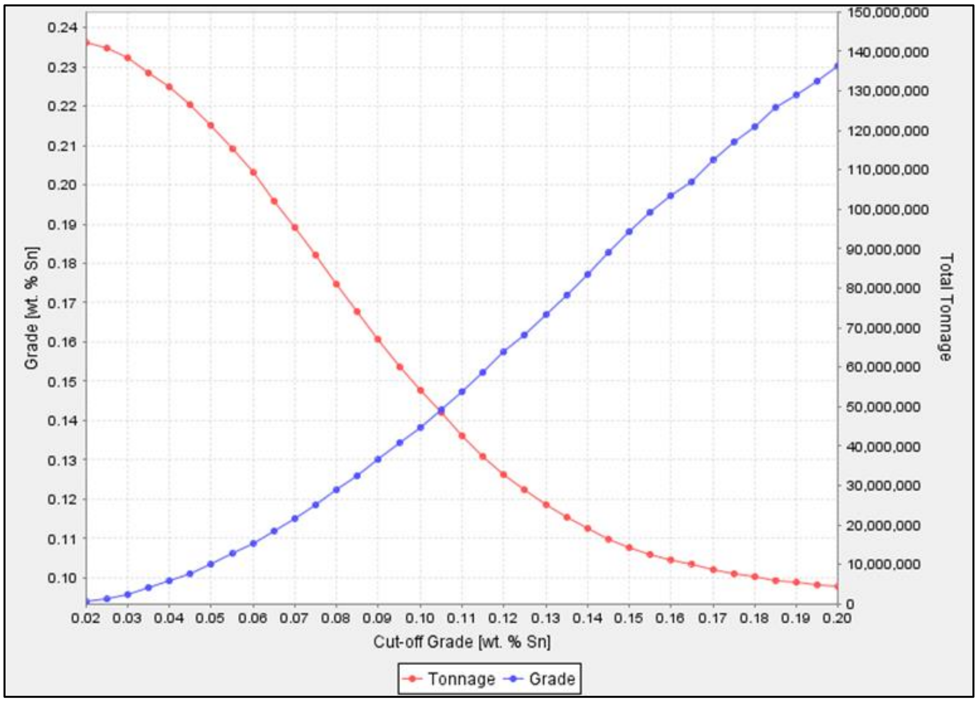


Figure 14.34
Grade-Tonnage Curve for Blocks in Measured and Indicated Categories in the Main Zone

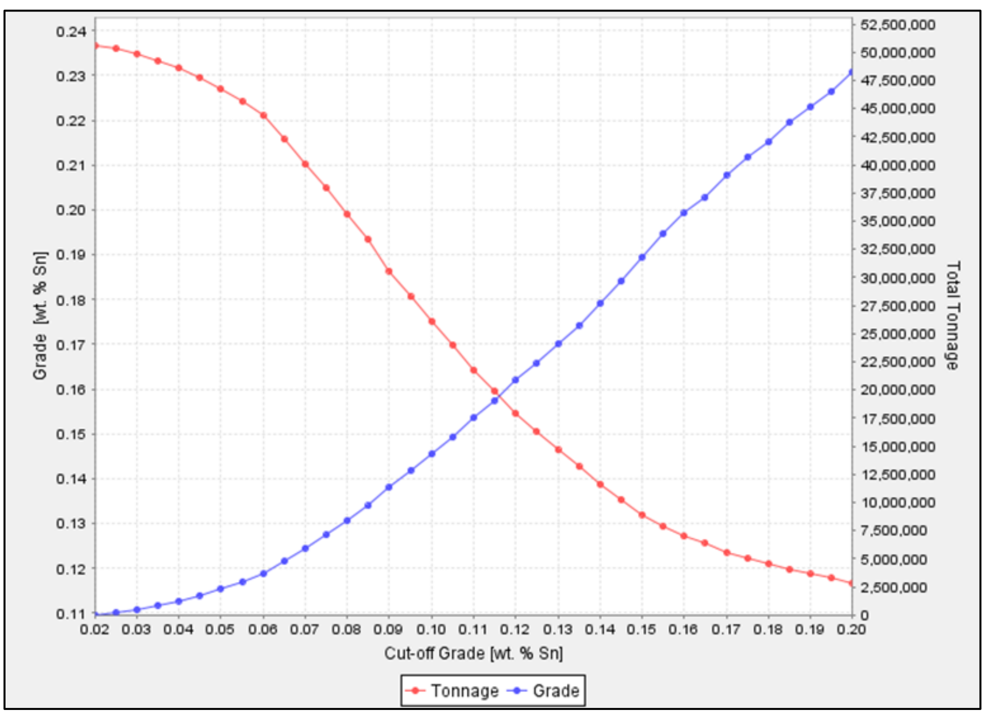


Figure 14.35
Grade-Tonnage Curve for Blocks in All Resource Categories in the Baby Zone

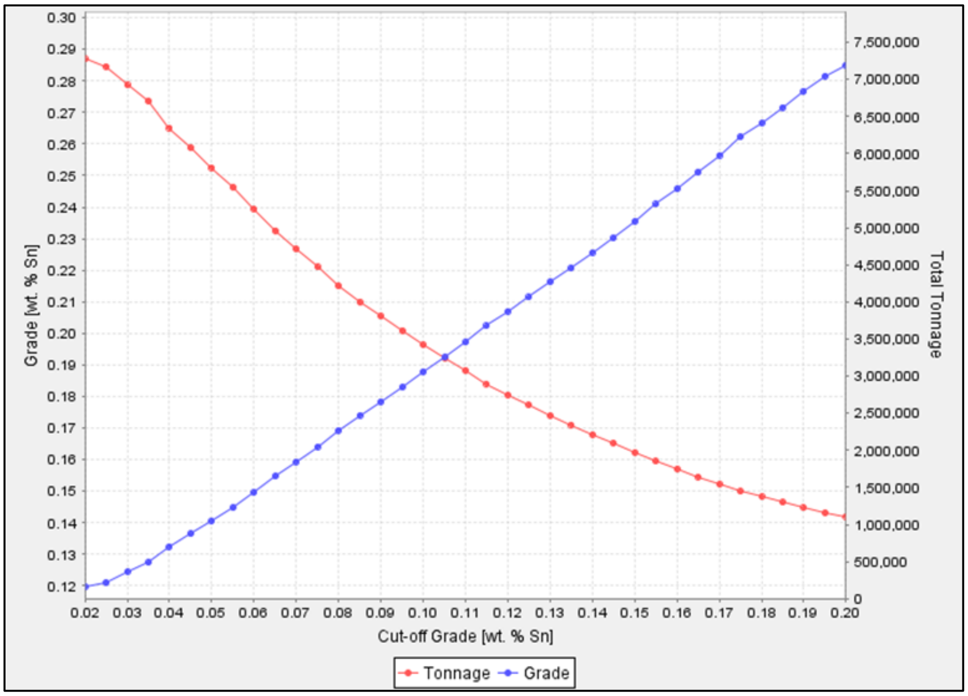
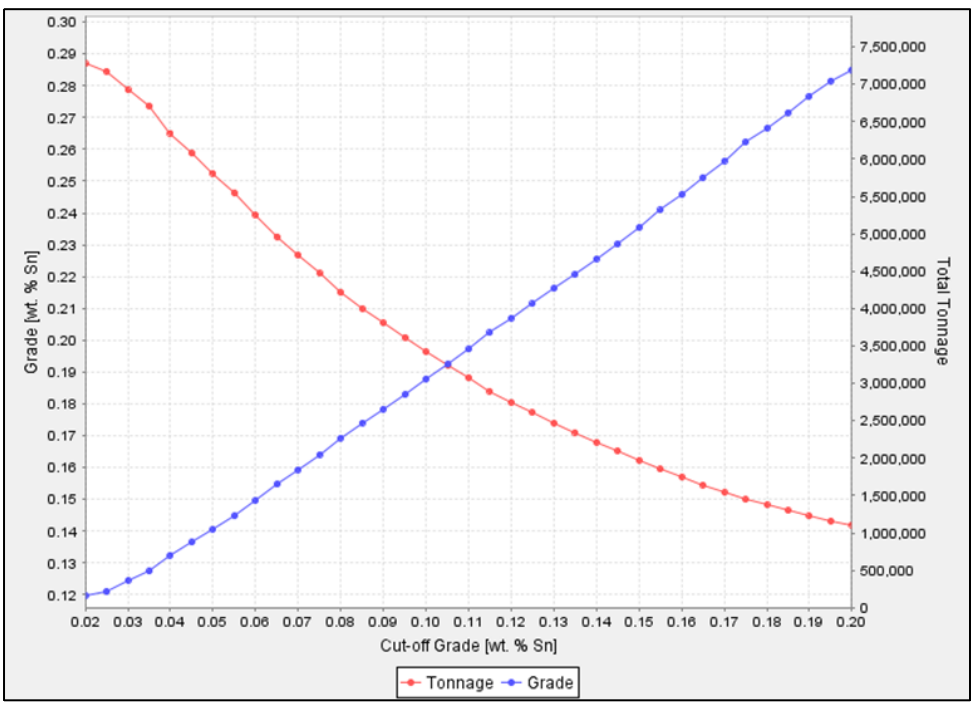


Figure 14.36
Grade-Tonnage Curve for Blocks in Measured and Indicated Categories in the Baby Zone



14.15 MINERAL RESOURCES BY ZONE

The mineral resource estimate presented here has been constrained by a preliminary pit design, utilizing a Whittle design developed during the Conceptual Economic Study presented in News Release 15-02 (25 February, 2015), which includes the Main and Baby Zones. The 2015 pit design utilized a bench height of 12 m and a slope angle no steeper than 50°. The main objective of applying the pit design from the previous study was to constrain the model from including resources that are presently considered unlikely to be mined but also to enable comparison with previous estimates. However, it should be noted that the present updated resource includes the results of the 2015 drill program on the property. The effect of applying the 2015 open pit design as a constraint to the resource is a significant reduction of the inferred resources around the margins of the Main Zone, which are, however, supported by geological models and composites from drill holes. For the Baby Zone, some indicated material is removed by applying the pit design because new drilling was performed in 2015, which was not considered for the 2015 pit design. The blocks in the pit design were selected based on a block percentage >50% within the volume between design and topography using vertical needling with an Integration Level of 10. The volumes considered for the current mine plan (see Section 16.0) are almost completely located within the 2015 pit design.

Mineralization at East Kemptville has historically been divided between the Main Zone and the Baby Zone to the southwest. An overview of the estimates for the Measured, Indicated and Inferred in situ (i.e., unmined) resources by zone is presented in Table 14.16 and Figure 14.37 and Figure 14.38 provide an overview of the resources in 3D. Mineral resources do not have demonstrated economic viability and their value may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other issues. All resource blocks were selected based on vertical needling with an Integration Level of 10 to capture blocks with >50% in situ granite.

Table 14.16
East Kemptville Mineral Resource Estimate, Main and Baby Zones, Based on Percentage Tin Cut-off Grade and Constrained by a Preliminary Pit Design, as at 7 May, 2018

Classification	Cut-off grade Sn (%)	Main Zone NE		Baby Zone		Total		
		Tonnes (Mt)	Sn (%)	Tonnes (Mt)	Sn (%)	Tonnes (Mt)	Sn (%)	Sn (tonnes)
Measured	0.08	0.40	0.173	0.22	0.241	0.61	0.197	1,200
	0.10	**0.38**	**0.177**	**0.20**	**0.251**	**0.58**	**0.203**	**1,200**
	0.12	0.32	0.188	0.19	0.259	0.51	0.214	1,100
Indicated	0.08	27.89	0.133	1.72	0.194	29.61	0.137	40,400
	0.10	**20.91**	**0.148**	**1.48**	**0.211**	**22.39**	**0.152**	**34,100**
	0.12	14.84	0.163	1.27	0.228	16.11	0.168	27,100
Measured + Indicated	0.08	28.28	0.134	1.93	0.199	30.22	0.138	41,700
	0.10	**21.29**	**0.148**	**1.68**	**0.216**	**22.97**	**0.153**	**35,100**
	0.12	15.16	0.164	1.46	0.232	16.62	0.170	28,200
Inferred	0.08	18.54	0.125	0.90	0.153	19.43	0.126	24,600
	0.10	**13.56**	**0.137**	**0.69**	**0.172**	**14.25**	**0.139**	**19,800**
	0.12	8.11	0.156	0.51	0.193	8.62	0.158	13,600

Notes:

1. CIM Definition Standards for Mineral Resources, 2014, were followed.
2. The independent Qualified Person for this Mineral Resource estimate is William Mercer, Ph.D., P. Geo. (Nova Scotia). The mineral resources are current as of May 7, 2018.
3. The mineral resource estimate is based on 194 drill holes totalling 21,456 m drilled between 1979 and 1991 by previous operators and 23 holes totalling 4190 m drilled by Avalon in 2014 and 2015.
4. Drill data were organized in Maxwell DataShed and for estimation purposes were transferred to the Geovia GEMS 6.8.1 software, wherein the block model was developed.
5. Resources were estimated by interpolating composites within block models of 24 m by 24 m by 12 m blocks in the Main Zone and 6 m by 6 m by 6 m in the Baby Zone. Interpolation used the Ordinary Kriging method.
6. In the Main Zone, Measured material was defined as blocks interpolated with a search ellipse with radii of 40x20x15 m using 18-36 samples, corresponding to 3-6 drill holes, indicated material with a 120x40x18 m search ellipse and the same number of samples, and inferred material with a 315x85x18 m search ellipse using 12-24 samples corresponding to 2-4 drill holes. In the Baby Zone, Measured material was defined as blocks interpolated with a search ellipse with radii of 30x20x8 m using 6-12 samples, corresponding to 3-6 drill holes, indicated material with a 48x33x12 m search ellipse and the same number of samples, and inferred material with a 95x65x24 m search ellipse using 4-8 samples corresponding to 2-4 drill holes (see Section 1.12 Resource Classification).
7. Prior to compositing, the assays were capped at 1% Sn, which corresponds to the 99th percentile of the tin assay data, reducing the length-weighted mean of the tin assays by 9.4%.
8. Mean density values of available data of 2.728 t/m^3 and 2.784 t/m^3 were used for the Main and Baby Zones, respectively.
9. The resource estimate has been constrained using the Whittle pit described previously (Avalon News Release 15-02, February 25, 2015).
10. Several possible cut-off grades are reported in this resource estimate. Based on past mining practice at East Kemptville, a cut-off grade of 0.1% Sn is reasonable and preliminary cost and revenue values at the time of estimation also suggest this is reasonable.
11. Mineral resources do not have demonstrated economic viability and their value may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other issues.

Figure 14.37
Overview of the Main Zone Block Model at a Cut-off Grade of 0.10 wt. % Sn in 3D
(The blocks are colour-coded by confidence category: measured – red, indicated – yellow, inferred – green)

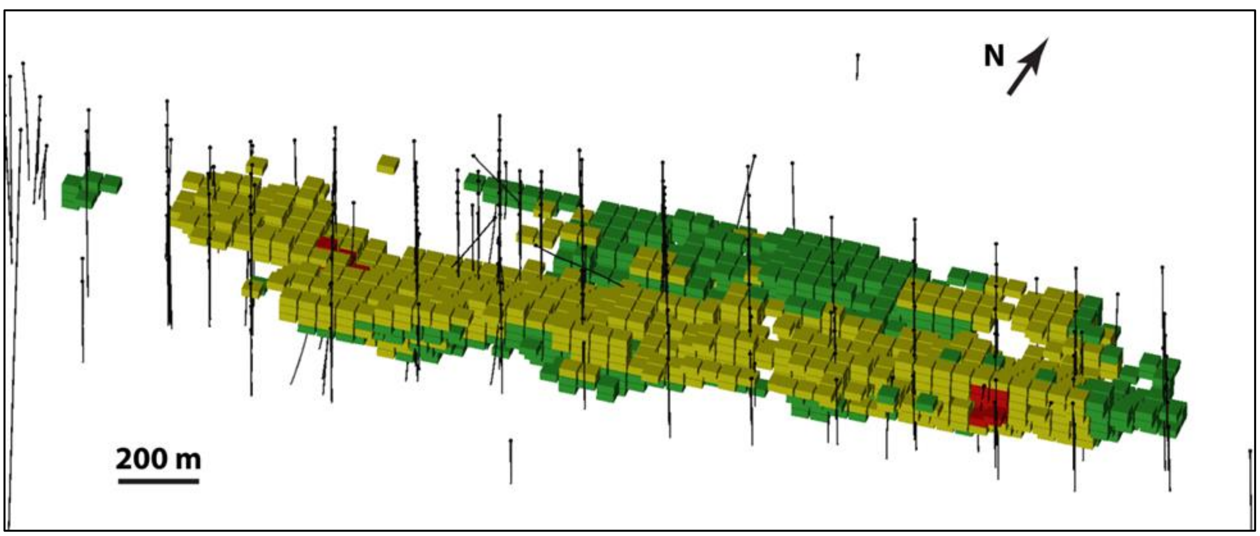
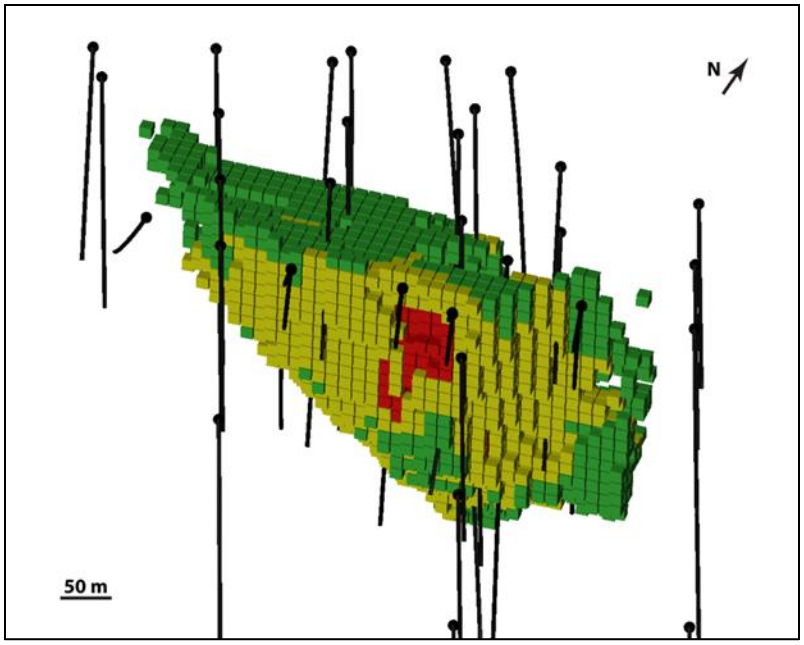


Figure 14.38
Overview of the Baby Zone Block Model at a Cut-off Grade of 0.10 wt. % Sn in 3D
(The blocks are colour-coded by confidence category: measured – red, indicated – yellow, inferred – green)



14.16 COMPARISON WITH HISTORIC RESOURCE ESTIMATES

The Feasibility Study presented by Rio Algom in 1983 (i.e., pre-mining) estimated geological resources of 66 Mt grading 0.155% Sn at a cut-off grade of 0.08% Sn for the East Kemptville deposit and was prepared using a cross-sectional area method (see Section 6.0 Table 6.1). Subtracting ~18 Mt of mined material during operations would suggest a remaining tonnage of 48 Mt, which is comparable with the current estimate. While viewed as reliable and relevant based on the information and methods used at the time, the historic resource of Rio Algom (1983) does not satisfy the requirements set out by NI 43-101. Sampling and assay methods used may have resulted in understatement or overstatement of the grade and/or tonnage and thus contained tin, copper and zinc. The extent of understatement or overstatement is unknown. No Qualified Person has undertaken sufficient work to classify the historic estimate as a current resource. Avalon is not treating the historical estimate as a current resource. However, it is known that at the time of production, it was found that the blast hole drill analyses gave more reliable prediction of short term mill feed grades.

Wardrop (2010) presented a Desktop Study for the East Kemptville Deposit, which included an in situ (i.e., un-mined) resource estimate that was based on the historic drill database, a 3D geological model of the granite, and performed in Datamine using a 20 x 20 x 12 m block model and Ordinary Kriging with multiple indicator ranges (see Table 14.17). This method spatially restricted the interpolation of elevated grades. However, the exact methods for the multiple indicators were not given. The search ellipse used had dimensions of 80 x 40 x 20 m and was enlarged by factors of 1.5 and 2 for the second and third passes, respectively. Compared to the current resource estimate, the Wardrop (2010) estimate is significantly more restricted. A detailed analysis indicates that this is not a result of the search ellipse dimensions, but of the spatial limiting of elevated grades during interpolation. Without the ability to assess the 3D distribution of the resource, the difference cannot be analyzed further. A common feature of the current and the Wardrop (2010) resource estimates is that both have a high sensitivity of their tonnages to the cut-off grade (see Table 14.16 and Table 14.17).

Table 14.17
Historical Mineral Resource Estimate of Wardrop (2010) for the Main and Baby Zones

Cut-off grade (wt. % Sn)	Measured + Indicated			Inferred		
	Mt	wt. % Sn	Tonnes Sn	Mt	wt. % Sn	Tonnes Sn
0.06	29.08	0.122	35,510	25.522	0.103	26,411
0.08	15.72	0.167	26,302	9.971	0.158	15,732
0.10	**11.70**	**0.195**	**22,867**	**6.319**	**0.199**	**12,567**
0.12	10.46	0.206	21,503	5.049	0.221	11,184
0.14	8.77	0.220	19,290	4.068	0.244	9,905
0.16	7.26	0.235	17,024	3.207	0.269	8,619
0.18	5.75	0.252	14,466	2.637	0.290	7,659
0.20	4.48	0.269	12,048	2.079	0.318	6,603
0.22	3.24	0.291	9,455	1.589	0.351	5,572
0.24	2.42	0.312	7,559	1.363	0.371	5,052
0.26	1.95	0.327	6,370	1.115	0.397	4,429
0.28	1.37	0.352	4,819	0.854	0.437	3,729
0.30	0.99	0.376	3,740	0.735	0.461	3,387

In 2014, Avalon presented an estimate for the remaining in situ resources of the East Kemptville deposit, which was based on new holes drilled by Avalon in 2014 and the historic drill hole data base (Table 14.18). The estimate used the contact between the granite and the metasediments as a boundary, a 5 x 5 x 3 m block size and was performed using the Inverse Distance Weighting Squared method with a spherical search ellipse of 100 m dimensions and combined with a localization of high-grade assays. Compared to the current estimate, the Avalon (2014) method yielded a more constrained indicated resource (~4.5 Mt less tonnage at a 0.1 wt. % Sn cut-off) with a higher grade (0.176 vs. 0.153 wt. % Sn, compare (Table 14.16 and Table 14.18). This difference is due to the prevention of interpolation between the 100 m drill section lines and the resulting localization of the 2014 resources around the drill section lines. Notably, the 2014 estimate significantly underreported the grade and tonnage of the mined volume.

At a cut-off of 0.1 wt. % Sn, the inferred resource of the 2014 estimate has a ~2.7 Mt larger tonnage than that of the current estimate and a higher grade. Owing to the different estimation methods, the grade-tonnage curves are significantly different between the current and the 2014 estimate. As is the case for the current estimate, the 2014 estimate also shows a high sensitivity to the applied cut-off grade and, despite a severe spatial restriction of the grade interpolation, yielded a much larger tonnage than the Wardrop (2010) estimate.

Table 14.18
Resource Estimate of Avalon (2014) for the Main and Baby Zones Combined.

Cut-off grade (wt. % Sn)	Indicated			Inferred		
	Mt	**wt. % Sn**	**Tonnes Sn**	**Mt**	**wt. % Sn**	**Tonnes Sn**
0.05	46.07	0.104	47,913	34.29	0.102	34,976
0.10	**18.47**	**0.176**	**32,507**	**16.95**	**0.148**	**25,086**
0.15	6.83	0.239	16,324	2.66	0.203	5,400
0.20	3.16	0.337	10,649	0.82	0.311	2,550
0.25	2.93	0.344	10,079	0.58	0.342	1,984

14.17 ADDITIONAL MINERAL RESOURCES

Additional mineral resources are available in various stockpiles located on site. The RAL closure plan of 1993 identified the low-grade stockpile situated to the immediate northwest of the Main Pit as having the following characteristics:

Estimated Tonnes	5.87 million	
Estimated Grades	Sn	0.106%
	Zn	0.106%
	Cu	0.068%

The estimated grades are close to the average resource grade and the low-grade stockpile may represent an additional source of feed to the processing facility. Additional measurement and sampling of the stockpile is required to confirm the historic tonnage and grade data.

14.17.1 Reporting Standards for Stockpiles

To quote the Estimation of Mineral Resources and Mineral Reserves (MRMR) Best Practice Guidelines adopted by CIM Council on November 23, 2003, for consideration of the resources contained in stockpiles, etc.:

> Additional guidance for reporting of MRMR estimates: Item (o): *"Broken mineralized inventories, as an example, surface and underground stockpiles, must use the same basis of classification outlined in the CIM Standards. Mineralized material being processed (including leaching), if reported, should be reported separately."*

As part of this study, an examination of what other companies have done to verify historic resource stockpiles was completed. There are very few descriptions of such work in the public realm, but the few that exist are useful as some guidance on what practice may exist in this area.

14.17.2 East Kemptville Waste Piles and Stockpiles

Mineral resources are present in various stockpiles located at the East Kemptville site, for which resource estimates are provided in the Rio Algom (RAL) Closure Plan (RAL, 1993) and which is quoted in Table 14.19: The footnotes to the table comprise information provided in the RAL Closure Report.

Table 14.19
Low Grade Stockpiles and Waste Piles Characteristics, Rio Algom (1993)

Dump	Tonnes (Mt)	Surface Area (ha)	Sample Number	Grade (%)		
				Sn	Zn	Cu
North Waste Dump	1.29	6.8	NWD-1	0.116	0.180	0.070
			NWD-2	0.096	0.120	0.050
			NWD-3	0.055	0.090	0.030
Measured[1]			Mean	0.089	0.130	0.050
South Waste Dump and Pad	2.82	19.4	SD-1	0.026	0.080	0.030
			SD-2	0.065	0.180	0.030
Measured[1]			Mean	0.046	0.120	0.030
Estimated[2] North and South			Mean	0.049	0.102	0.035
Low Grade Stockpile	5.87	22.7	LG-1	0.188	0.090	0.080
			LG-2	0.057	0.090	0.040
			LG-3	0.140	0.160	0.060
			LG-4	0.076	0.060	0.090
			LG-5	0.078	0.130	0.050
Measured[1]			Mean	0.106	0.106	0.068
Estimated[2]			Mean	0.091	0.153	0.058

[1] Measured by RAL in samples collected from surface piles in 1991.
[2] Estimated by RAL from more than 1,000 sample analyses from blasthole cuttings.

In the original Feasibility Study, RAL classified the rock into “High Grade”, which was above a cutoff grade of 0.12% Sn and “Low Grade” which was the remaining materials above a cutoff of 0.08% Sn (see Section 6.0 Table 6.1).

Cut off grade during operations (1985 to 1990) were reported to be:

- Material at better than 0.12% Sn, delivered to the mill.
- Low grade mineralization at 0.08 to 0.119% Sn, delivered to Low Grade Stockpiles.
- Waste rock below 0.08% Sn, delivered to waste piles.

Grades were determined by blasthole sampling.

14.17.3 Grade and Tonnage Verification

The Low-Grade Stockpile reaches an elevation of about 113 m above sea level, and the surrounding land is at about 93 m above sea level, so the thickness of the main part of the pile is about 20 m. Measured on Google Earth, the Low-Grade Stockpile has a perimeter of about 1,500 m and an area of about 135,000 m^2. At a density of 1.6 t/m^3, this would give a very approximate tonnage of about 4.5 Mt.

In order to verify more accurately the tonnage present in the stockpiles and waste dumps, a volume estimate was completed using Minesight software and the following:

- The original 1983 topography prior to mining.
- The topography from the 1992 topographic survey at the close of mining.

A density (SG) of 1.6 t/m^3 was then applied as this was considered reasonable from past experience with estimating resources in stockpiles and dumps. The results are summarized in Table 14.20. The Avalon estimate of tonnage is within 5.5% overall of that quoted by RAL for all stockpiles and dumps.

Table 14.20
Volume and Tonnage Estimates, Low Grade Stockpile

Area	Avalon			RAL	Difference
	Volume (m^3)	SG	Tonnes	Tonnes	
North Waste Dump	810,078	1.60	1,296,125	1,290,000	-0.5%
Low Grade Stockpile + South Waste Pad	4,653,489	1.60	7,445,582	5,870,000	
South Waste Dump and Pad				2,820,000	
South Waste Dump	432,505	1.60	692,008		
Total			**9,433,715**	**9,980,000**	**5.5%**

The sampling and grade verification process is described in Section 11.1.3.

14.17.4 Low Grade Stockpile Mineral Resources

On the basis of its investigation, Avalon considers that the Low-Grade Stockpile may be reported as an Inferred Mineral Resource as summarized in Table 14.21 (Reference: Annual Information Form 20F, Avalon Advanced Materials Inc., August 31, 2016. Table EK2. Accessible on Avalon Website or www.Sedar.com). The accepted metal grades are the average of the RAL and Avalon surface sampling.

Table 14.21
Low Grade Stockpile Estimated Inferred Mineral Resource

Category	Tonnes (Mt)	Grade (%)		
		Sn	Zn	Cu
Inferred	5.87	0.112	0.100	0.61

Notes:
1. This estimate is as of 16 November 2015.
2. CIM Definition Standards 2014 were followed for mineral resources.
3. The independent Qualified Person for this Mineral Resource estimate is Donald H. Hains, P.Geo., of Hains Engineering Company Limited.
4. Resources were estimated by examination of historical RAL data and Avalon's 2015 sampling of the Low-Grade Stockpile.
5. Mineral resources do not have demonstrated economic viability and their value may be materially affected by environmental, permitting, legal, title, socio-political, marketing or other issues.

Additional measurement and sampling of the stockpile, especially with depth, is required to confirm the historic tonnage and grade data to a higher category of resource such as Indicated or Measured.

15.0 MINERAL RESERVES ESTIMATES

No mineral reserves have been estimated for the East Kemptville tin Project.

16.0 MINING METHODS

16.1 INTRODUCTION

Avalon plans to engage a locally (Eastern Canada) based mining contractor to mine material from the Rio Algom Ltd. (RAL) legacy stockpile, the Baby Zone pit and the Main Zone pit. This contractor will be responsible for supplying, operating and maintaining all mining equipment, trucks and mining related infrastructure. Avalon's cost estimate has only budgeted for emptying the pits of both water and any sludge that may have settled on the bottom.

The mining will re-start as a continuation of the historic open pit operation using a conventional drill and blast process and conventional truck and shovel methods for material movement. Mill feed will be hauled to a small (24 h) stockpile and then reclaimed into the crusher rock box by a front-end loader. The mill treatment rate is set at 2,208 t/d (806,000 t/y) starting initially using only stockpiled material, which will then be supplemented by new feed mined from the Baby Zone Pit during production Years 2 and 3. Thereafter, mining will take place from the southern portion of the Main Zone Pit in Years 3 through 7, followed by production from two pits located in the northern portion of the Main Pit Zone in Years 7 through 14, after which the operation will be fed with the remainder of the legacy stockpile through Years 15 to 19. During Years 2 through 14 when the majority of mill feed is sourced from the pits, approximately 6 weeks' worth of material will be scheduled from the legacy pile in order to (a) incrementally reduce the environmental liability and (b) maintain mill productivity in the event significant operational delays are encountered in the mine.

For this PEA, the life-of-mine (LOM) open pit mineable plant feed material within the conceptual pit designs is 9.22 Mt, inclusive of Measured, Indicated and Inferred resources, with a total waste movement of 3.24 Mt for an average stripping ratio of 0.35:1. With the inclusion of an additional 5.87 Mt of Inferred resources from the RAL legacy stockpile, the mine life is extended to 19 years.

16.2 PIT OPTIMIZATION

Two rotated resource block models were prepared by Avalon at different block resolutions (6 m x 6 m x 6 m for the Baby Zone, and 24 m x 24 m x 12 m for the Main Zone). The two models were merged to create a single functional model suitable for mine planning and pit optimization at a resolution of 6 m x 6 m x 6 m. The economic parameters used as inputs into Datamine NPV Scheduler® are summarized in Table 16.1.

Table 16.1
Pit Optimization Criteria East Kemptville Tin Project

Parameter	Unit	Value
Mining Cost (Mill Feed)	CAD/t. mined	4.70
Mining Cost (Waste)	CAD/t. mined	4.70

Parameter	Unit	Value
Legacy Stockpile Rehandling	CAD/t. moved	1.25
Process Cost (Concentrator)	CAD/t. mill feed	7.85
Process Cost (Sulphide Flotation)	CAD/t. mill feed	0.63
G&A	CAD/t. mill feed	1.54
Pit Slope	degrees	See Figure 16.1
Processing Recovery (Sn)	%	60.0
Metal Price (Sn)	USD/t	20,656
Treatment Charge	CAD/t.conc	455
Transportation	CAD/t.conc	225
Exchange Rate	USD to CAD	1.30

Nested Lerchs-Grossman pit shells were generated in Datmine NPV Scheduler® using incremental price factors or "revenue factors." Next, using a minimum mining width assumption of 40 m, practical phased mining sequences within each nested shell were generated to produce a series of conceptual production schedules. The phasing and extraction sequence generating the maximum NPV was based on the revenue factor 0.70 pit which was exported to Vulcan to guide the pit designs.

16.3 PIT DESIGN

16.3.1 Parameters

The pit slope angles used in the pit optimization are shown in Figure 16.1. The schematic is reproduced from the 2009 Wardrop study, based on the recommended pit slope angles from the report "A study on the "Fast Wall" Stability at Rio East Kemptville Open Pit Tin Mine, East Kemptville, Nova Scotia", dated January 31, 1992. As per this study, inter-ramp angles up to 56° could be maintained with a double-bench interval of 24 m, provided controlled blasting and other practices such as drainage ditches, berm clearing, and scaling of faces are put in place. The conceptual pit design parameters detailed in Table 16.2 reflect the following envisaged fleet which is typical of small-volume contractors available in Nova Scotia:

- 1C390 Excavator
- 3 to 4 773/775 Haul trucks
- 1 D8 Dozer
- 1 L8 Drill
- C349 Excavator
- 140M Grader
- 966 Loader
- Water Truck
- Fuel Truck

Production benches will be drilled and blasted in 12-m bench intervals and excavated in 6-m flitches. The 6-m benches will be stacked up to 24 m.

Figure 16.1
Pit Slope Inter-Ramp Design Angles

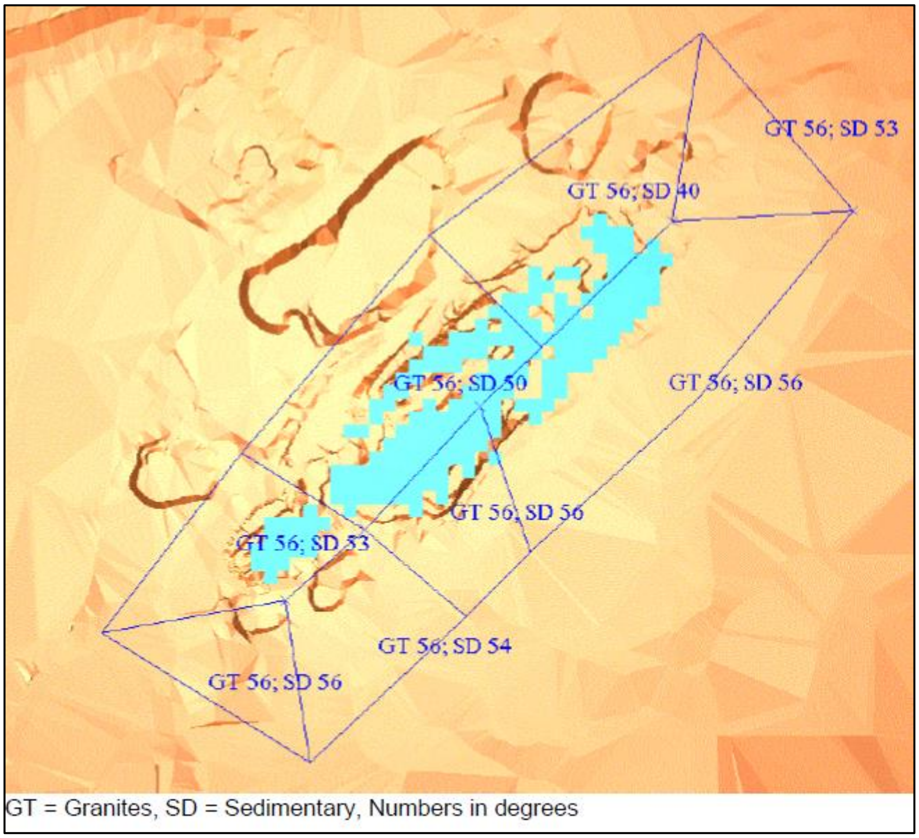


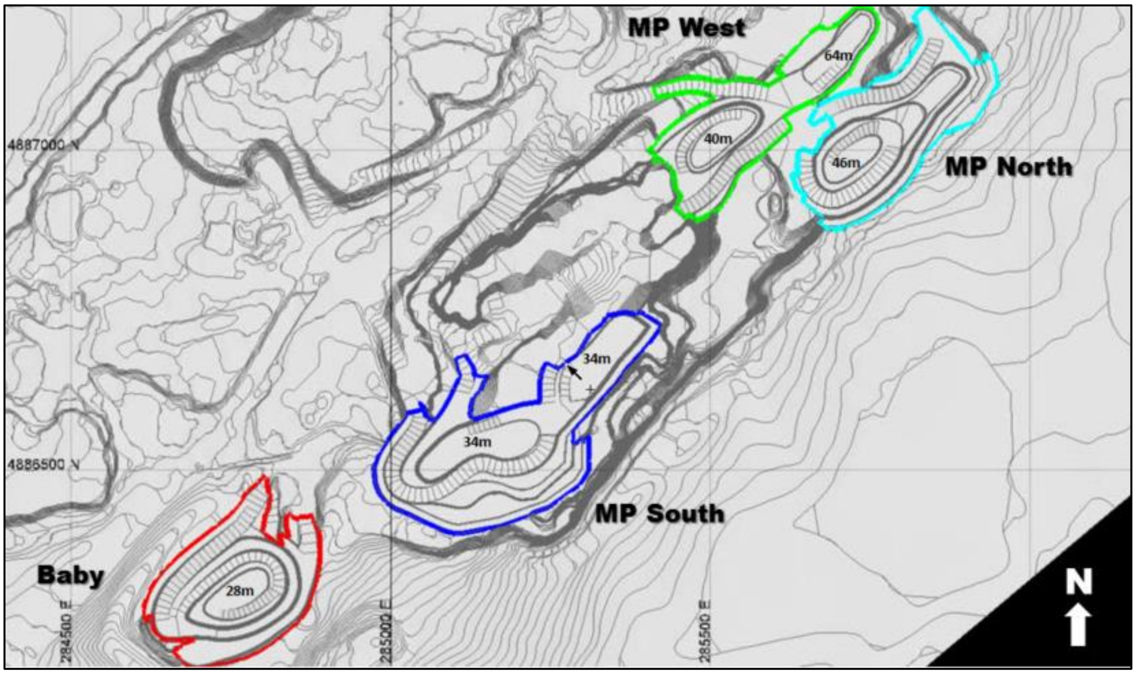
Table 16.2
Conceptual Pit Design Parameters

Design Parameters	Units	Comments
Bench Heights	12 m	Mined in 6 m flitches
Bench Face Angle	75°	
Catch Berm Width (Baby Pit Zone)	12 m	Highwalls in metasediment
Catch Berm Width (Main Pit Zone)	10 m	Highwalls in granite
Benches between Catch Berms	2	
Inter-ramp Angle	56°	
Overall Pit Slopes	48°	
Ramp width, Two -way traffic	22 m	
Ramp width, One-way traffic	14 m	Final benches
Ramp Gradient, Two-way traffic	10%	
Ramp Gradient, One-way traffic	12%	Final benches

16.3.2 Life-of-Mine Design

The ultimate pit design is illustrated in Figure 16.2. Access was developed to facilitate material routing through the pit to minimize mill feed and waste hauls.

Figure 16.2
Ultimate Pit Design (Based on Revenue Factor 0.70 Pit)



16.4 PIT EXTRACTION AND BACKFILL SEQUENCE

The extraction and backfill sequence, as illustrated in Figure 16.2 to Figure 16.9 requires minimal waste rehandling and ensures adequate land-bridges between pits are maintained to prevent flooding of ongoing operations. Waste rock piles were designed to an angle of repose of 36° constructed in 4 m, 6 m or 8 m lifts as required to establish critical land-bridge elevations. Ramp widths of 22 m were designed at grades ranging from 10% to 12%. The available fill capacities for waste and tailings are summarized in Table 16.3.

Figure 16.3
Years 2 through 6 – Baby and Main Pit South Waste Rock Shipped to Waste Zone A

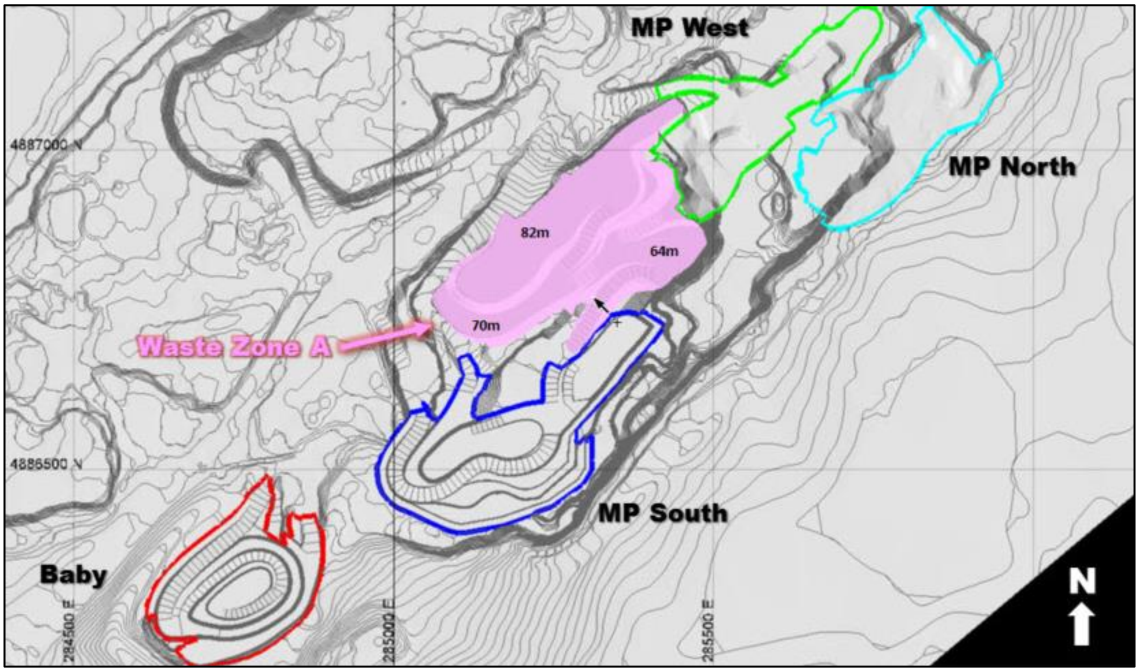


Figure 16.4
Years 7 through 10 – Main Pit West Waste Rock Shipped to Waste Zone A, B, and Lift 70 of Zone C

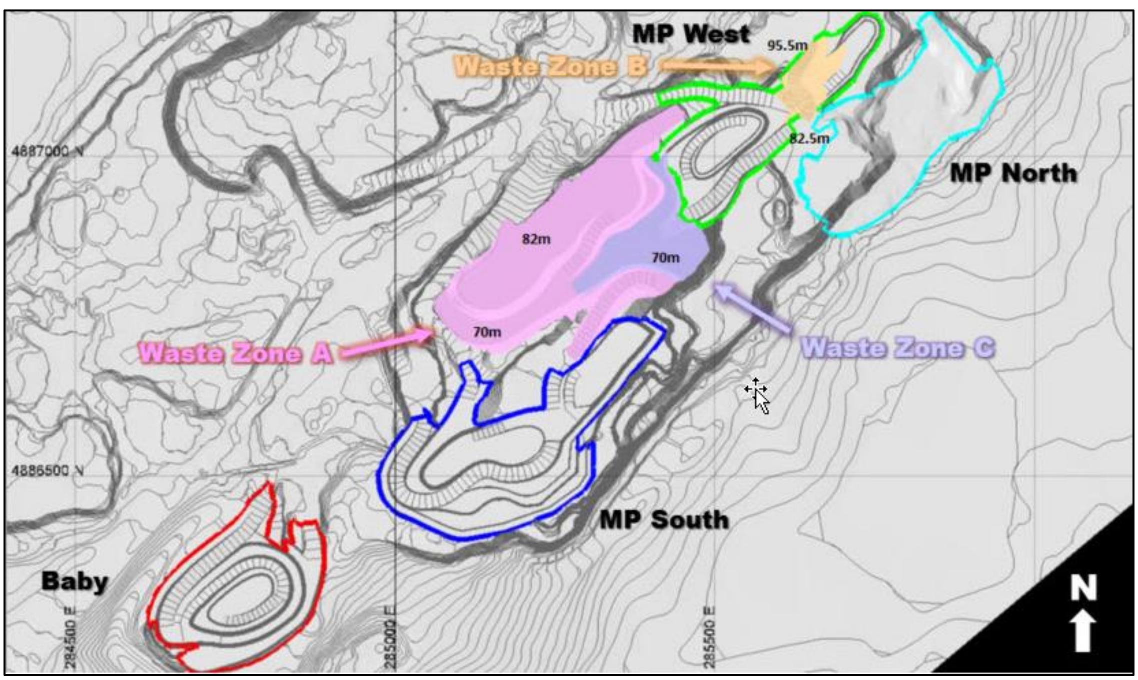


Figure 16.5
Years 11 through 14 – Main Pit North Waste Production to Waste Zone C and D

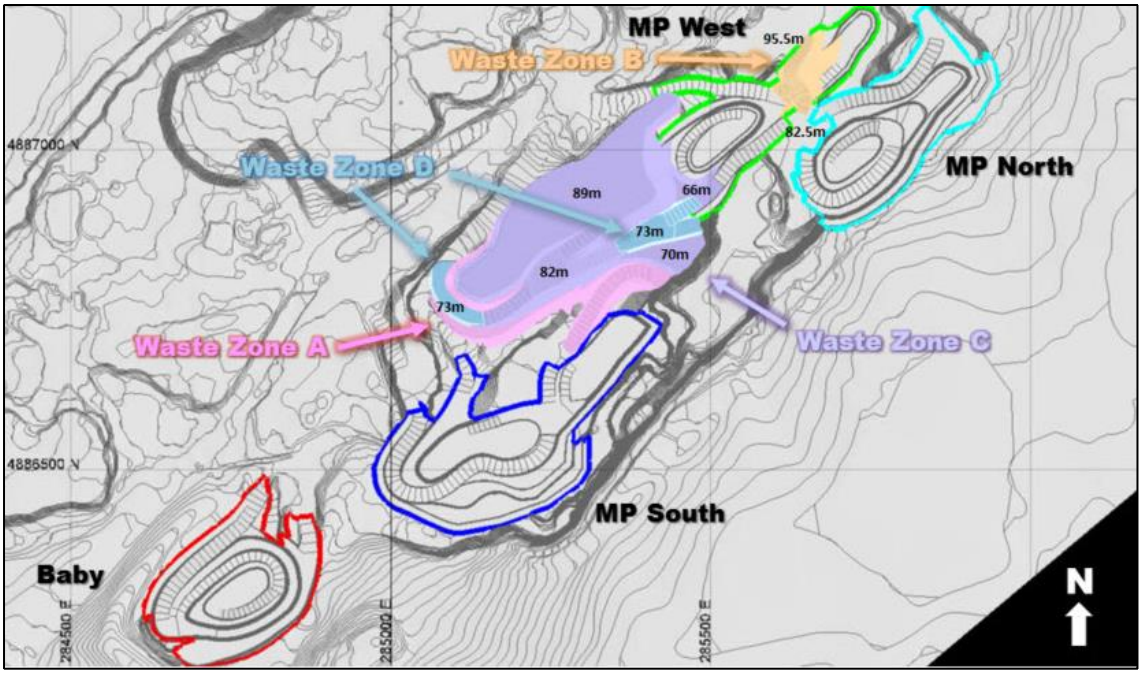


Figure 16.6
Tailings Backfill Elevations – Years 4(Q4) through 12(Q3)

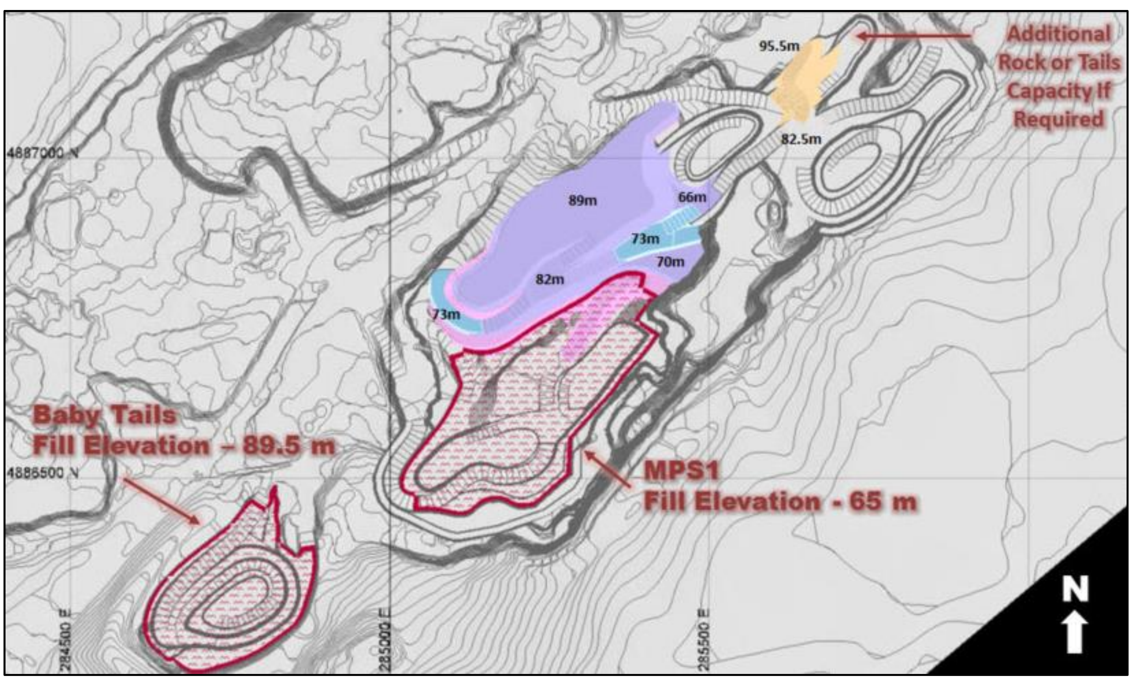


Figure 16.7
Tailings Backfill Elevation – Years 12(Q4) through 13

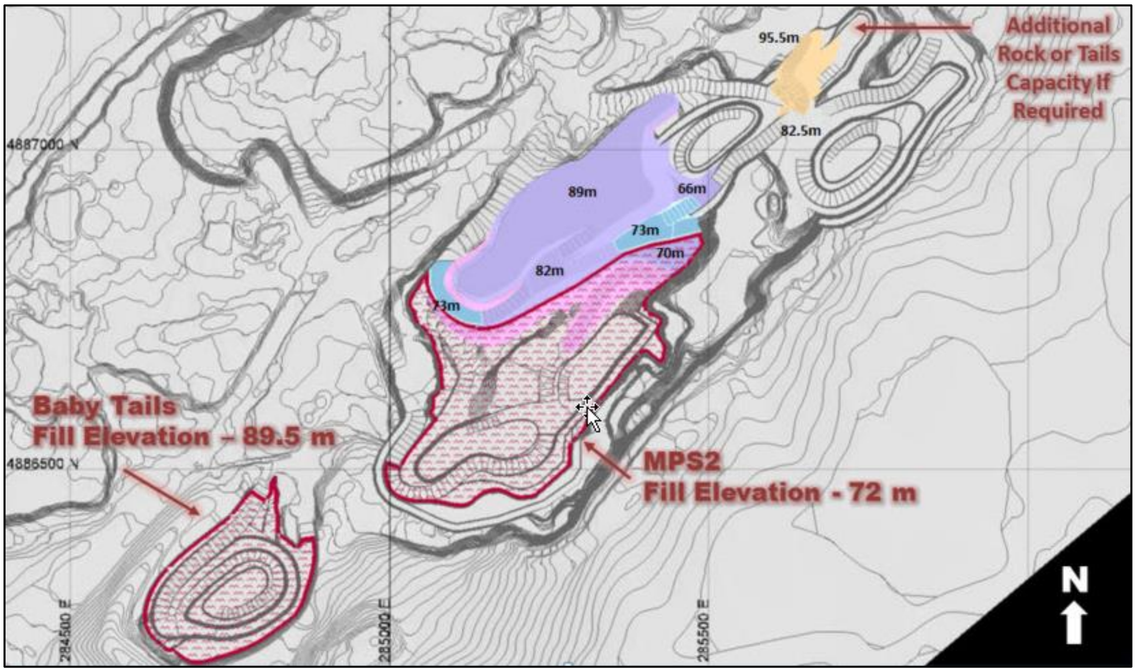


Figure 16.8
Tailings Backfill Elevation – Years 14 through 17(Q3)

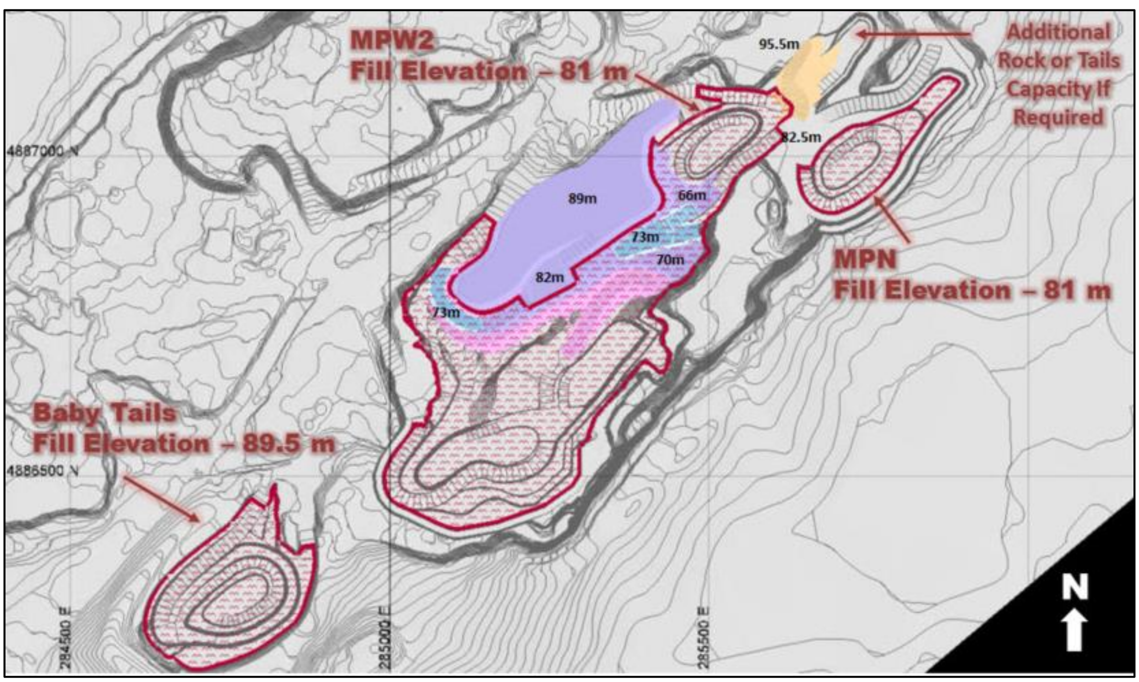


Figure 16.9
Final Backfill Elevation for Tailings and Other Legacy Waste Piles

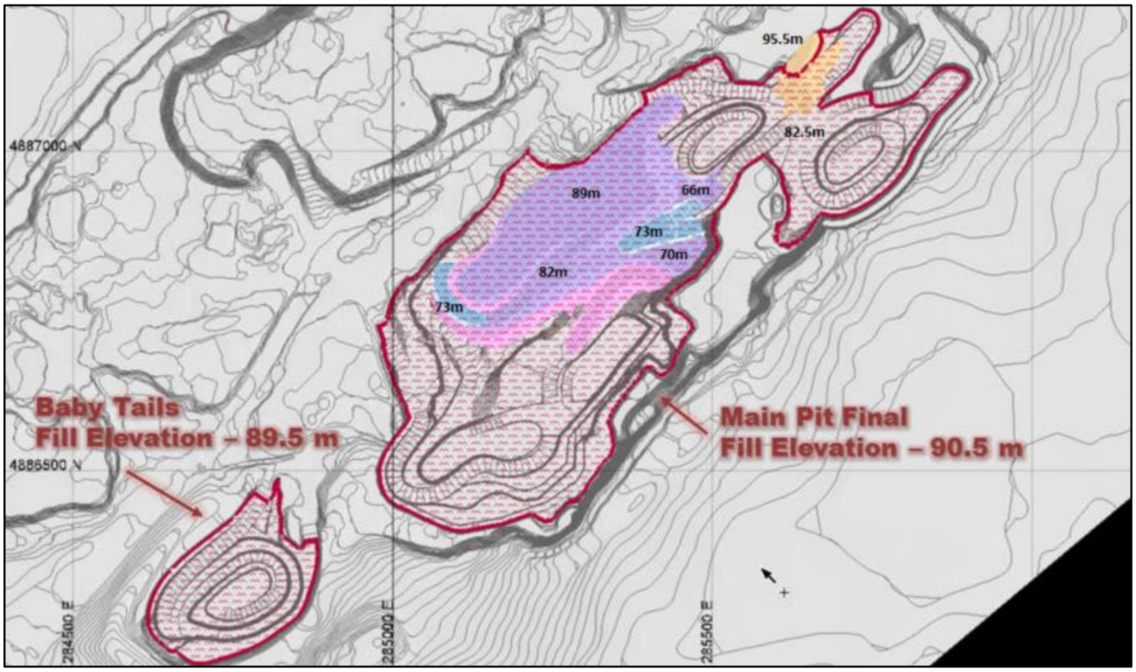


Table 16.3
Available Storage Capacity for Waste Rock and Tailings

Storage Location	Fill Elevation (m)	Available Capacity (m^3)
Waste Zone A	82	1,415,989
Waste Zone B	93	77,620
Waste Zone C	89	512,449
Waste Zone D	73	22,546
Main Pit South 1 (MPS1)	65	1,268,380
Main Pit South 2 (MPS2)	72	612,949
Main Pit West (MPW)	81	1,644,299
Main Pit North (MPN)	81	401,327
Remaining MP Capacity When Pit Production Ceases	22	2,500,461

16.5 PRE-PRODUCTION SCHEDULE

The Pre-production schedule is a function of the time required to dewater and remediate the Baby Zone Pit to a condition suitable to resume mining and the anticipated construction period required for the plant. Other pre-production activities will include upgrades to the tailings facility and the water treatment facility to handle the additional water originating from the Baby Pit. Dewatering of the Main Pit Zone will continue through the first month of Year 2.

16.6 MINE PRODUCTION SCHEDULE

The proposed production plan is presented in Table 16.4. Phasing and production planning was developed to meet the following operational and environmental objectives:

- To support Avalon's strategy for a "walk away" closure plan that will isolate all potentially acid generating waste rock and tailings from oxygen and/or water, allow for the storage of all waste and tailings in the excavated pits that will be flooded when operations cease.
- To the extent practicable, avoid the need for temporary ex-pit storage of waste and minimize re-handling costs.
- Allow for an 18-month start up period to build the process plant, begin dewatering of both pits and prepare the roads and tailings facility for operation.
- Create sufficient capacity in Main Pit South as soon as possible to accommodate tailings after the TMF and Baby Pit tailings storage areas have reached design limits.
- Maintain internal access between the three Main Zone pits to minimize waste rehandling and provide shortened hauls for plant feed, thereby reducing the greenhouse gas emissions footprint of the site remediation plan.
- Facilitate safe and productive simultaneous mine and backfill operations in the Main Zone by maintaining a minimum freeboard of one metre in the tailings repositories to manage water after storm events.
- Provide for a freeboard allowance of one metre in the final backfilled pit configuration.
- Maximize NPV within all constraints defined above.

Key milestones for the conceptual Project development are described below:

Year -1: Dewatering of the Baby Pit and Main Pit begins, along with site preparation for construction of plant.

Year 1: Final dewatering and removal of sludge in the Baby Pit takes place during Q1 and Q2. Shipment from the legacy stockpile begins in Q3, with process tailings directed to the TMF. Dewatering of the Main Pit continues and will carry over into the first month of Year 2.

Year 2: Mine operations in the Baby Pit begin supplemented by feed from the legacy stockpile. Waste rock is shipped to its final destination to the west of Main Pit South (Waste Zone A, Figure 16.3) and tailings are directed to the TMF. Any overburden material suitable for use in future revegetation work will be set aside.

Year 3: Baby Pit production ceases and Main Pit South mining begins, supplemented with legacy stockpile feed. Waste rock from both pits is shipped to Waste Zone A and process tailings are sent to the TMF.

Year 4-6: Production feed is sourced from Main Pit South, supplemented with the RAL legacy stockpile. Waste rock is shipped to Waste Zone A. Once the TMF capacity is reached in the last quarter of Year 4, tailings will be directed to the Baby Pit. Beginning in Year 6, mill feed from the Main Pit will be processed through a sulphide flotation circuit to produce clean tailings to cap the TMF.

Year 7: Main Pit South production ceases and Main Pit West mining begins, supplemented with legacy stockpile feed. A portion of the waste rock from Main Pit West will be temporarily staged as required to create a fill ramp in Year 9 (Waste Zone B, Figure 16.4) to haul mill feed excavated from the lower benches. Sulphide tailings are directed to the Baby Pit, with low-sulphide tailings directed to capping operations.

Year 8-10: Production feed sourced from Main Pit West is supplemented with the RAL legacy stockpile. Stockpiled waste rock from the upper benches of Main Pit West is used to construct a fill ramp needed to haul mill feed from the lower benches in year 9 and 10. Remaining waste in is shipped to the final lift of Waste Zone A and the first lift (70 m elevation) of Waste Zone C (Figure 16.4). Low-sulphide tailings are directed to capping operations and sulphide tailings are deposited in the Main Pit South repository (MPS1, Figure 16.6).

Year 11-14: Production feed is sourced from Main Pit North, supplemented with the RAL legacy stockpile. Waste rock is shipped to Waste Zone C and D (Figure 16.5). Sulphide tailings continue to be directed to Main Pit South, with Year 11 being the final year of low-sulphide tailings production used for surface capping of the TMF. In Year 12, additional capacity in Main Pit South becomes available that will be sufficient for tailings deposition through Year 13 (Figure 16.7). In Year 14, tailings deposition will be directed to the Main Pit West area.

Year 15-19: Mill production comes from the RAL stockpile for the remainder of the Project life. Tailings deposition continues in Main Pit West and Main Pit South areas through Year 16 (Figure 16.8). Beginning in Year 17, tailings are directed to the Main Pit North pit area, after which tailings will continue to be deposited in the Main Pit for the remainder of the schedule (Figure 16.9). If evenly distributed, the estimated tailings volume in the Main Pit will approach an elevation of 87.5 m, leaving approximately 0.9 Mm^3 of additional capacity that may be used to reach the final allowable fill elevation of 90.5 m (one metre below the final water table). Potential uses for this capacity are discussed in Section 20.3.

16.7 MINE INFRASTRUCTURE

Mine infrastructure requirements are discussed in Section 18.0, Project Infrastructure.

Table 16.4
Production Schedule

Year Number	LOM TOTAL	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Legacy Stockpile																				
Plant Feed Tonnes	5,870,000	403,000	103,820	74,670	46,675	89,854	77,075	107,348	110,311	141,589	83,496	94,121	98,120	92,958	139,632	841,466	841,466	841,466	841,466	841,466
Sn%	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112	0.112
Baby Pit																				
Waste Tonnes	1,662,727		1,402,490	260,237																
Plant Feed Tonnes	1,249,947		702,180	547,767																
Sn%	0.197		0.212	0.178																
Strip Ratio	**1.33**		**2.00**	**0.48**																
Main Pit West																				
Waste Tonnes	196,806							12,346	86,168	58,264	40,028									
Plant Feed Tonnes	2,218,343							135,739	695,689	664,411	722,504									
Sn%	0.163							0.16	0.16	0.16	0.17									
Strip Ratio	**0.09**							**0.09**	**0.12**	**0.09**	**0.06**									
Main Pit North																				
Waste Tonnes	836,664											749,570	81,308	5,419	367					
Plant Feed Tonnes	2,799,168											711,879	707,880	713,042	666,368					
Sn%	0.151											0.145	0.152	0.151	0.155					
Strip Ratio	**0.30**											**1.05**	**0.11**	**0.01**	**0.00**					
Main Pit South																				
Waste Tonnes	541,286			347,687	188,452	4,148	1,000	0												
Plant Feed Tonnes	2,950,872			183,563	759,325	716,146	728,925	562,913												
Sn%	0.166			0.118	0.164	0.165	0.173	0.178												
Strip Ratio	**0.18**			**1.89**	**0.25**	**0.01**	**0.00**	**0.00**												
Plant Feed Grade Sn%	**0.144**	**0.112**	**0.199**	**0.158**	**0.161**	**0.159**	**0.167**	**0.166**	**0.151**	**0.153**	**0.166**	**0.141**	**0.147**	**0.147**	**0.148**	**0.112**	**0.112**	**0.112**	**0.112**	**0.112**
Total Plant Feed Tonnes	**15,088,329**	**403,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**806,000**	**841,466**	**841,466**	**841,466**	**841,466**	**841,466**

17.0 RECOVERY METHODS

17.1 INTRODUCTION

The metallurgical process for the East Kemptville Project is based on the testwork that is described in Section 13.0 and comprises the mineral separation and recovery of a tin concentrate with a target grade of 55 % Sn. A small portion of the copper, zinc, iron and indium will be collected into a sulphide concentrate which will be appropriately disposed of in the tailings facility unless a buyer for the material is found.

Production is based initially on the processing of 806,000 t/y of stockpiled material, but after Year 1, there will be an introduction of higher grade mineralization mined from the Baby Zone Pit. In Year 3, there will be the introduction of plant feed material from the Main Zone Pits. Construction of a bulk sulphide flotation circuit to remove sulphides from the plant tailings will begin in year 5, with sulphide-free tailings production beginning in Year 6 through Year 11. These low-sulphide "clean" tailings will be filtered and used as "capping material" for sealing the old tailings facility.

17.2 PROCESS FLOWSHEET DEVELOPMENT

The original East Kemptville metallurgical flowsheet comprised conventional process operations such as crushing, two-stage grinding, bulk sulphide flotation followed by differential Cu/Zn flotation and gravity recovery for tin. Such a flowsheet was determined to be uneconomic at current metal prices. Consequently, Avalon developed a simpler process focused on the production of a single tin concentrate. Data gathered from both the SGS UK and Met-Solve metallurgical test programs (see Section 13.0) along with historical information from previous operations and operating personnel was reviewed and used as the basis for developing the flowsheet.

Avalon's objective is to construct a simple plant with as few unit operations as possible that is focused purely on the recovery of a saleable tin concentrate. It is acknowledged that this approach will result in a lower than possible metal recovery, but it is believed that the low costs associated with such an approach will out-weigh any potential tin losses.

The primary consideration has been to produce a suitably sized milled product from a single stage grinding system and simultaneously prevent over-grinding of the plant feed and resultant production of very fine cassiterite grains which would be difficult to recover. This is achieved by incorporating a gravity recovery circuit within the grinding/classification circuit and also with the use of classifying screens rather than cyclones in the circuit.

17.3 PROCESS DESIGN BASIS

The PEA is based on the following assumptions derived from the testwork results:

- 806,000 t/y of stockpiled mineralized material will be fed to the concentrator at a rate of 100 t/h.

- Target primary grind P_{80}= 80 Microns.
- The tin gravity concentrate grade of 55% Sn and tin recovery to concentrate of ~60%.
- Plant availability of 91.3% for the concentrator (8,000 h/y operating time).
- Preliminary tin recovery will be by centrifugal concentrators with shaking tables used to produce the final product.
- Concentrate cleaning will include magnetic separation and flotation to remove iron, copper and zinc sulphides.

17.4 PROCESS DESCRIPTION

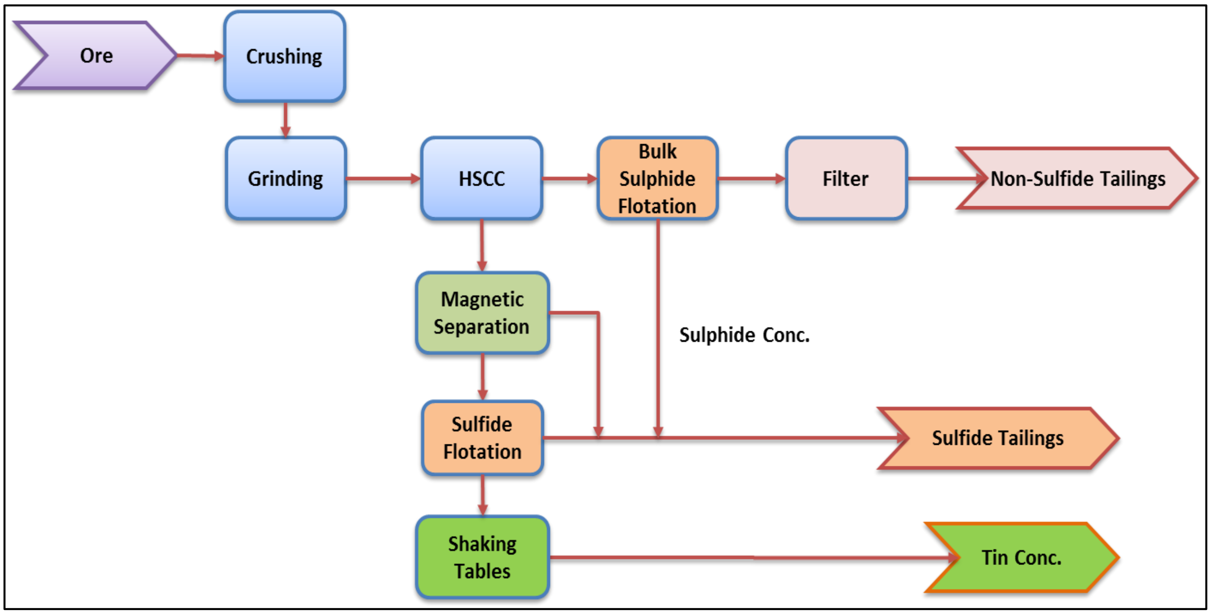
Process flow diagrams (PFDs) showing the selected process are included at the end of this section.

17.4.1 Overview

The 100 tonne per hour operation to treat the mineral resources at East Kemptville consists of several conventional processes to produce a tin concentrate. The plant feed undergoes crushing in a three staged crushing circuit, milling and classification, and is then put through a series of gravity circuits using high-speed centrifugal concentrators (HSCCs), magnetic separation and flotation to remove the metal sulphide before going through a series of shaking tables. A bulk sulphide flotation circuit is also included in Year 6 of the operation to remove sulphides from the gravity tailings (See Figure 17.1).

Benign flotation tailings will be filtered and used for capping the tailings facility. The bulk sulphides concentrate removed from the gravity tailings will be combined with the sulphide concentrate from the tin gravity circuit and stored under a cover of water to prevent oxidation.

Figure 17.1
Simplified Flowsheet



17.4.2 Crushing Circuit

The crushing circuit (see Figure 17.2) consists of a three staged crushing circuit and a series of screens for classification and magnets for tramp iron removal. Run-of-mine (ROM) material is fed at a rate of 200 t/h to a vibrating grizzly feeder (with 50 mm openings) by a front-end loader. Material greater than 400 mm is retained ahead of this feeder by a static grizzly screen and broken with a hydraulic rock breaker.

Material coarser than 50 mm is fed to a jaw crusher and then combined with feeder undersize before being classified at 50 mm by a vibrating primary screen. Primary screen oversized material is further crushed in a secondary cone crusher the product from which is combined with primary screen undersize and then screened at 8 mm by a secondary screen. Oversize (+8 mm) reports to a tertiary cone crusher before being recycled back to the secondary screen. The -8 mm product from the secondary screen is conveyed and stored in the fine crushed ore bin.

17.4.3 Milling and Classification Circuit

The grinding and classification circuit (see Figure 17.3) consists of a ball mill, a coarse screen and 6 “stacks” of fine screens (80 µm). The ball mill discharge is pumped to the coarse classification screen to remove +0.5 mm material which is recycled to the mill. The -0.5 mm material is pumped to a 4-way distributor, 2 outlets of which feed continuous primary HSCC concentrators (space will be provided for installation of up to a further 2 concentrators in the future if necessary). Tailings from the primary HSCC concentrators are combined with the other 2 outlets from the distributor, along with recycled gravity table tails and cleaner 1 and 2 HSCC concentrate tails and pumped to the stack of -80 µm fine screens. Screen oversize is recycled to the ball mill while screen undersize is feeds the main HSCC gravity circuit.

17.4.4 Gravity Circuit

Screen underflow from the milling circuit feeds a 2-way distributor with each outlet feeding continuous rougher HSCC concentrators in the gravity circuit (see Figure 17.3). Tailings from the rougher concentrators are pumped to tailings. Concentrates from the rougher concentrators are combined with primary HSCC concentrates and fed to a primary cleaner HSCC concentrator. Primary cleaner concentrate feeds a secondary cleaner concentrator, the concentrate from which is pumped to a sulphide removal circuit. Tailings from cleaner concentrators are combined and recycled to the fine screens in the milling circuit.

17.4.5 Sulphide Removal Circuit

The gravity concentrate sulphide removal circuit (see Figure 17.3) comprises a low intensity magnetic separation (LIMS) unit and a small flotation circuit to remove the entrained sulphides in the centrifugal concentrate. Gravity concentrate feeds a rougher LIMS, to scavenge entrained tin.

Non-magnetics from both magnetic separators are combined and sent to a small, sulphide flotation circuit where sulphide flotation reagents, such as Copper Sulphate, Xanthate and MIBC, will be added during a two-stage conditioning process. Flotation rougher concentrate is pumped to a small cleaner circuit to remove entrained tin with the final sulphide concentrate then being combined with the magnetics stream and pumped to the final tailings facility where they will be contained under water to prevent oxidation.

17.4.6 Tin Shaking Tables Circuit

The non-sulphide material is pumped to the tin shaking table circuit (see Figure 17.4) which consists of a dewatering cyclone and a series of 4 shaking tables used to produce a final tin concentrate containing ±55% tin. The dewatering cyclone thickens the slurry to ±40% solids which feeds a four-way distributor ahead of 4 shaking tables operating in parallel. Final concentrates from the four shaking tables are combined and fed into a holding tank before being pumped to filters for final dewatering. Table middlings are recycled back to the table feed whilst table tailings are recycled back to the milling/classification circuit. The dewatered tin concentrate filter cake will contain approximately 10% moisture and will be collected in large one tonne super-sacs before being containerized and shipped to potential customers.

17.4.7 Bulk Sulphide Flotation

Gravity circuit tailings will be pumped to the bulk sulphide flotation circuit (see Figure 17.5) comprising a single bank of rougher flotation cells where a sulphide concentrate will be recovered. Potassium amyl xanthate (PAX) will be used as a collector with MIBC/TF250 added as frother. Provision is also included in the design and costing for the addition of lime (for pH control) and copper sulphate although the Met-Solve testwork indicated these will probably not be required.

The circuit is designed to maximize sulphide removal in order to achieve the targeted <0.05% S in the final tailings. The tailings after sulphide removal will be filtered and the cake transported by truck to the tailings facility where it will be used for capping as part of the tailings facility remediation process.

17.4.8 Tailings and Sulphide Concentrates Storage

The tailings treatment circuit (see Figure 17.6) consists of a holding tank and a series of pumps. There are two sections to tailings and sulphide concentrate treatment, the first processes the tin tailings and the second handles the sulphide concentrate from the sulphide removal circuits.

Tailings from the gravity circuit will initially be pumped to the existing tailings facility and distributed appropriately across the facility (and below the current water line). Once the existing facility is filled, the material will be deposited inside one of the existing, mined out pits.

Sulphide/magnetics concentrate from the tin concentrate cleaning circuits will have a separate holding tank and also be pumped to the existing tailings facility (initially) but to a single, sub-aqueous location.

At the end of Year 5 of operations, the bulk sulphide flotation circuit will be installed to reduce sulphur content of the gravity tailings to <0.05%. The sulphide concentrate will be deposited under water inside one of the abandoned/mined out pits. Once the tailings facility is fully capped, this large flotation circuit will no longer be used (See Environmental Section 20.0 for more details).

17.4.9 Flotation Reagents

Flotation reagents will be stored, mixed and pumped to specific addition points within the process using dedicated variable speed dosing pumps (see Figure 17.7). Some of the reagents will arrive on site in bulk and some will be in drums. The reagents include the following:

- Potassium Amyl Xanthate (PAX) – Collector.
- Methyl Isobutyl Carbinol (MIBC) – Frother.
- Lime- pH modifier (only used if required).
- Copper Sulphate- Activator (only used if required).

17.5 METALLURGICAL ACCOUNTING

Mechanical weightometers will be installed on the conveyors to the jaw crusher feed as well as the feed to the ball mill to both measure and control feed rates onto the conveyors.

Mass flow systems and/or sampler facilities will be installed on the following process streams:

- Jaw crusher product conveyor (weightometer).
- Crushed material to the ball mill (weightometer).
- Manual sampling of feed.
- Mill circuit product (automatic sampler).
- Final plant tailings (flowmeter, densitometer and automatic sampler).
- Feed to sulphide removal circuit (access for manual sample).
- Sulphide concentrate (access for manual sample).
- Tin shaking table concentrate (automatic sampler).
- Tin shaking table tailings (automatic sampler).

Samples will be taken several times per hour (frequency will vary depending on sample) and eight-hour composites will be sent to the laboratory for analysis.

17.6 PLANT SERVICES

17.6.1 Compressed Air

The processing plant will have a small compressor to generate both plant air and clean/instrument air (air which has been filtered and dried).

17.6.2 Raw Water

Raw water will be obtained from the existing supply and piped to a new storage tank. Fresh water from this tank will be used to provide gland service water, reagent make-up water and final concentrate filter wash water. Potable water also sourced from existing facilities will be used only for ablution and human consumption.

17.6.3 Process Water

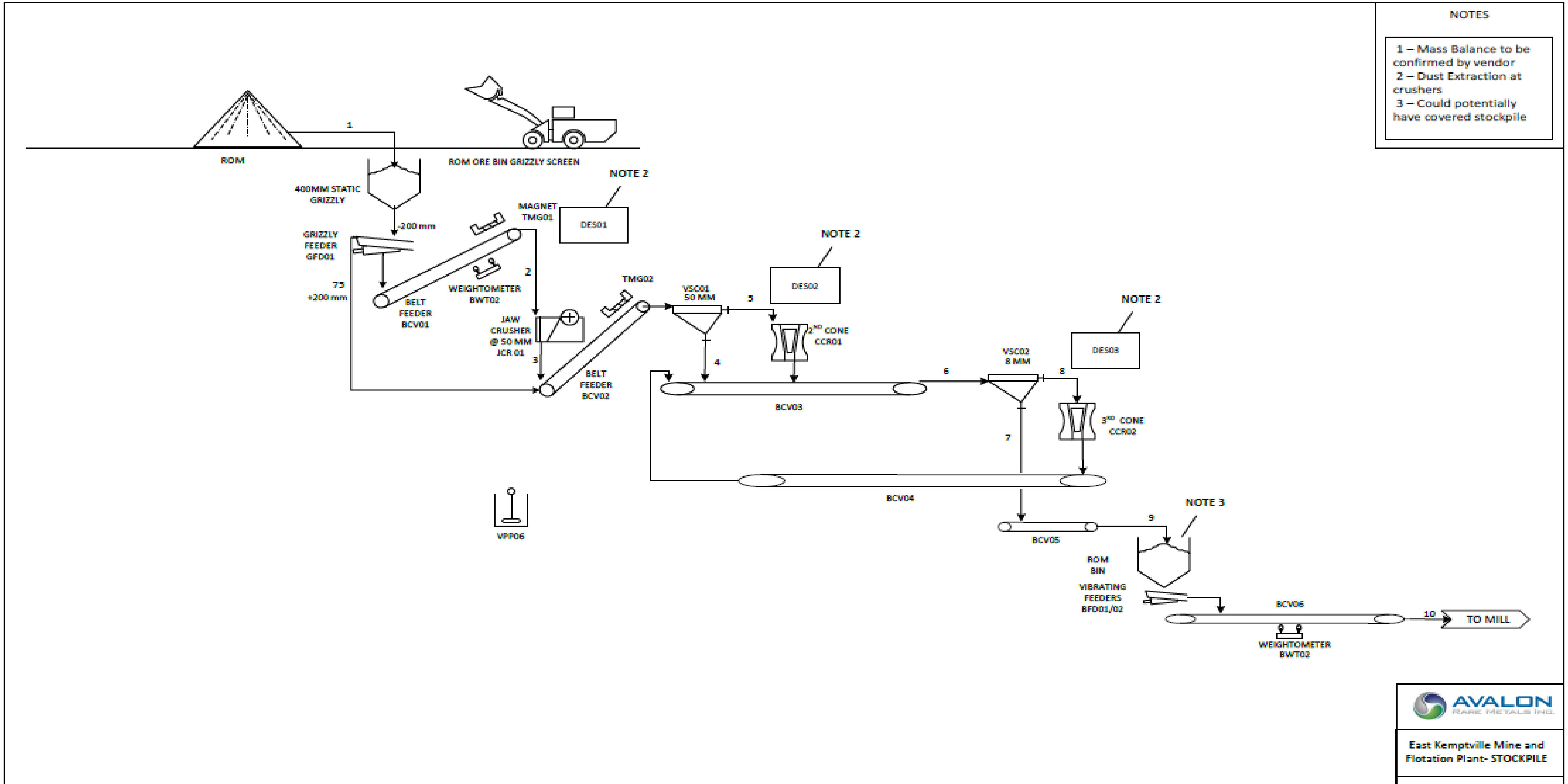
Process water will be used for the following duties:

- Comminution (mill dilution and screen sprays).
- Gravity concentrator circuits (flush water).

- Shaking Tables (wash water).
- Sulphide flotation circuits (launder sprays).

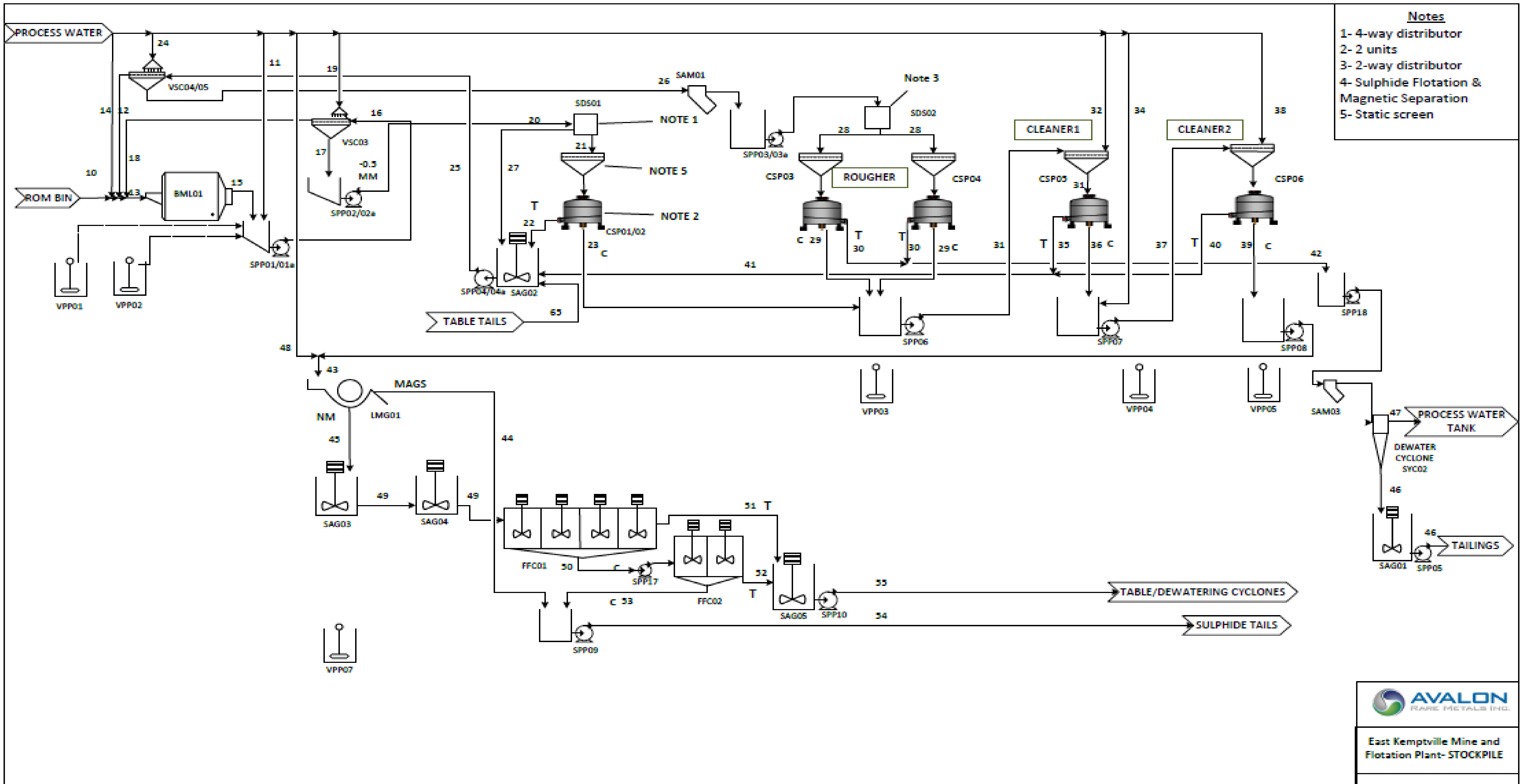
Process water will be recycled from the tailings treatment facility to both these circuits to minimize fresh water consumption.

Figure 17.2
Flowsheet and Mass Balance for the Crushing Circuit



Stream No	UNITS	1	2	3	4	5	6	7	8	9	10	75
MASS - SOLIDS	tph	200	160	160	80	120	333	200	133	200	100	40
MASS - SOLUTION	tph	10.5	8.42	8.42	4.21	6.32	17.5	10.5	7.02	10.5	5.26	2.11
MASS - SLURRY	tph	211	168	168	84.2	126	351	211	140	211	105	42.1
PERCENT SOLIDS (wt/wt)	%	95	95	95	95	95	95	95	95	95	95	95

Figure 17.3
Flowsheet showing Milling, HSCC Concentrator and Sulphide Removal Circuits

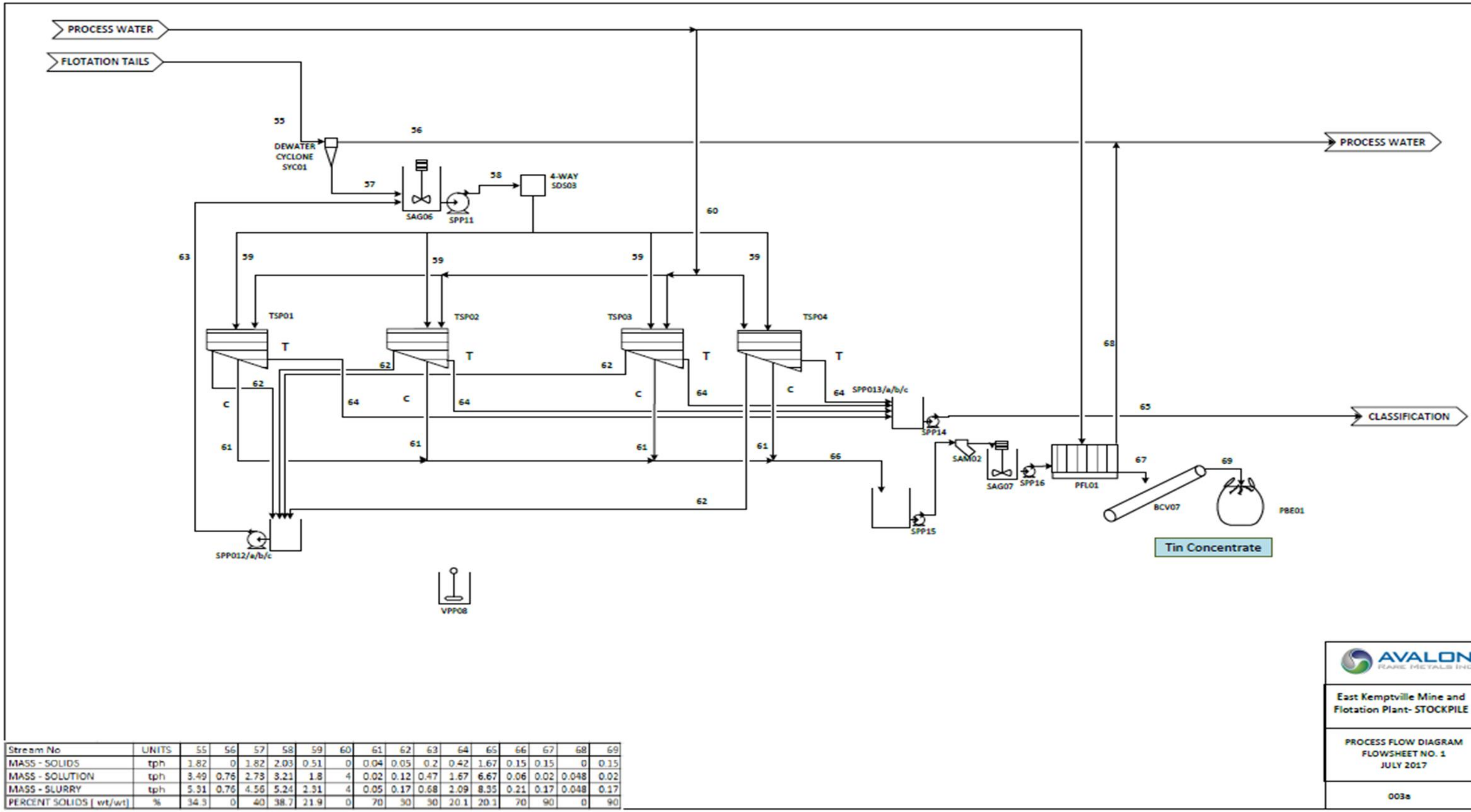


Stream No	UNITS	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28
MASS - SOLIDS	tph	100	0	146	351	0	351	351	246	105	0	246	123	98.3	24.6	0	270.3	124	123	62.2
MASS - SOLUTION	tph	5.26	201	7.68	18.5	132	150	351	365	5.54	20	365	183	158	24.6	20	445	437	183	219
MASS - SLURRY	tph	105	201	154	370	132	501	702	611	111	20	611	306	256	49.1	20	715	561	306	281
PERCENT SOLIDS [wt/wt]	%	95	0	95	95	0	70	50	40.2	95	0	40.2	40.2	38.3	50	0	37.8	22.1	40.2	22.1

Stream No	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46
MASS - SOLIDS	12.4	49.7	49.4	0	49.4	0	39.5	9.89	9.89	0	1.98	7.91	47.5	99.5	1.98	0.06	1.92	99.5
MASS - SOLUTION	12.4	206	49.4	24.7	74.2	4.94	64.3	9.89	14.8	0	1.98	12.9	77.1	412	3.67	0.04	3.63	99.5
MASS - SLURRY	24.9	256	98.9	24.7	124	4.94	104	19.8	24.7	0	3.95	20.8	125	512	5.65	0.1	5.55	199
PERCENT SOLIDS [wt/wt]	50	19.4	50	0	40	0	38.1	50	40	0	50	38.1	38.1	19.4	35	60	34.6	50

Stream No	47	48	49	50	51	52	53	54	55	65
MASS - SOLIDS	0	0	1.92	0.19	1.73	0.1	0.1	0.16	1.8	1.67
MASS - SOLUTION	313	1.69	3.63	0.29	3.35	0.14	0.14	0.18	3.5	6.67
MASS - SLURRY	0	1.69	5.55	0.48	5.07	0.24	0.24	0.34	5.3	8.35
PERCENT SOLIDS [wt/wt]	0	0	34.6	40	34	40	40	45.8	34	20.1

Figure 17.4
Flowsheets Showing Shaking Table and Tin Concentrate Dewatering Circuits



Stream No	UNITS	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69
MASS - SOLIDS	tph	1.82	0	1.82	2.03	0.51	0	0.04	0.05	0.2	0.42	1.67	0.15	0.15	0	0.15
MASS - SOLUTION	tph	3.49	0.76	2.73	3.21	1.8	4	0.02	0.12	0.47	1.67	6.67	0.06	0.02	0.048	0.02
MASS - SLURRY	tph	5.31	0.76	4.56	5.24	2.31	4	0.05	0.17	0.68	2.09	8.35	0.21	0.17	0.048	0.17
PERCENT SOLIDS (wt/wt)	%	34.3	0	40	38.7	21.9	0	70	30	30	20.1	20.1	70	90	0	90

Figure 17.5
Flowsheets Showing Bulk Sulphide Flotation Circuits

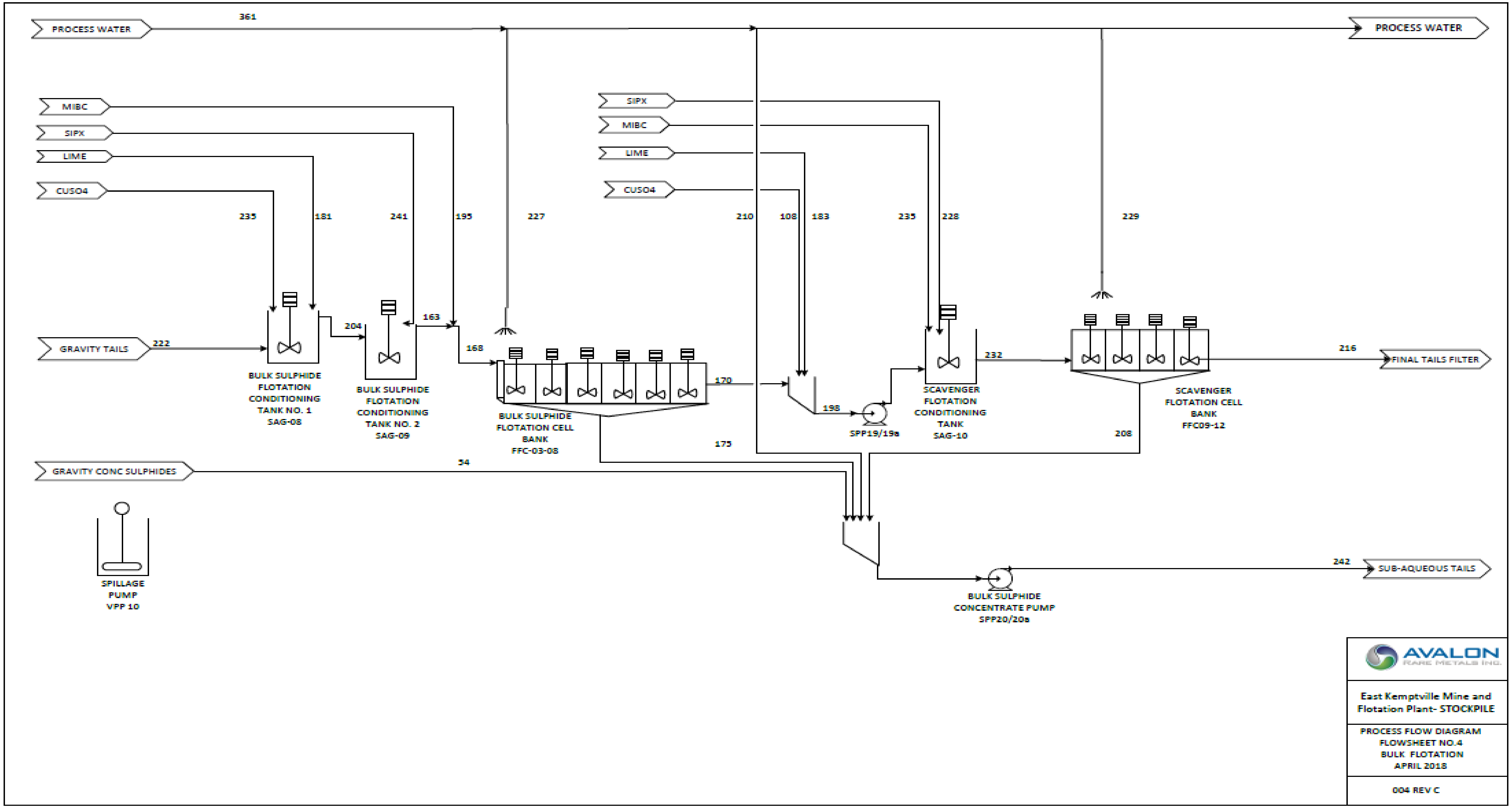


Figure 17.6
Flowsheets Showing Tailings Area

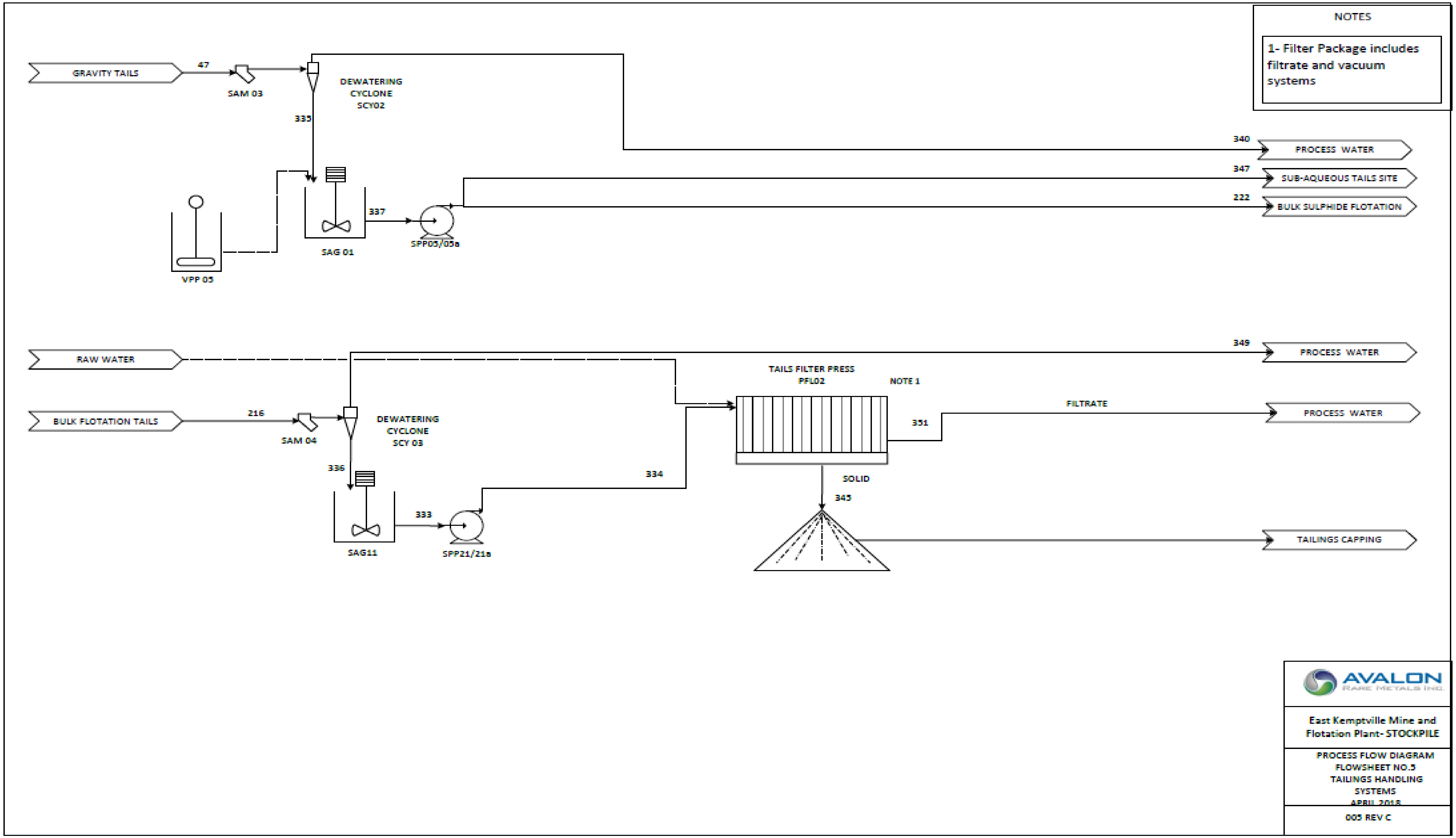


Figure 17.7
Flowsheets Showing Reagents Area

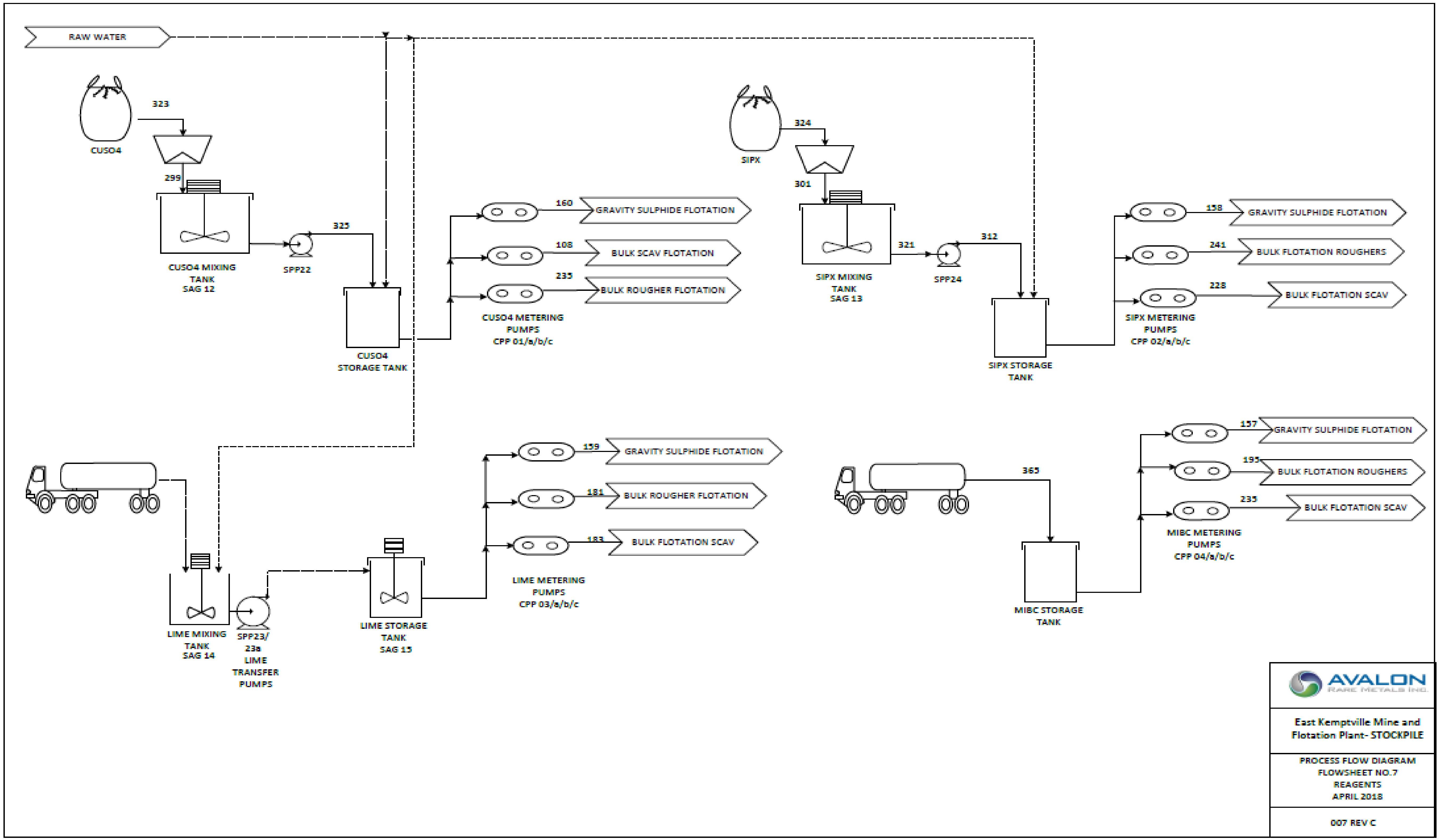
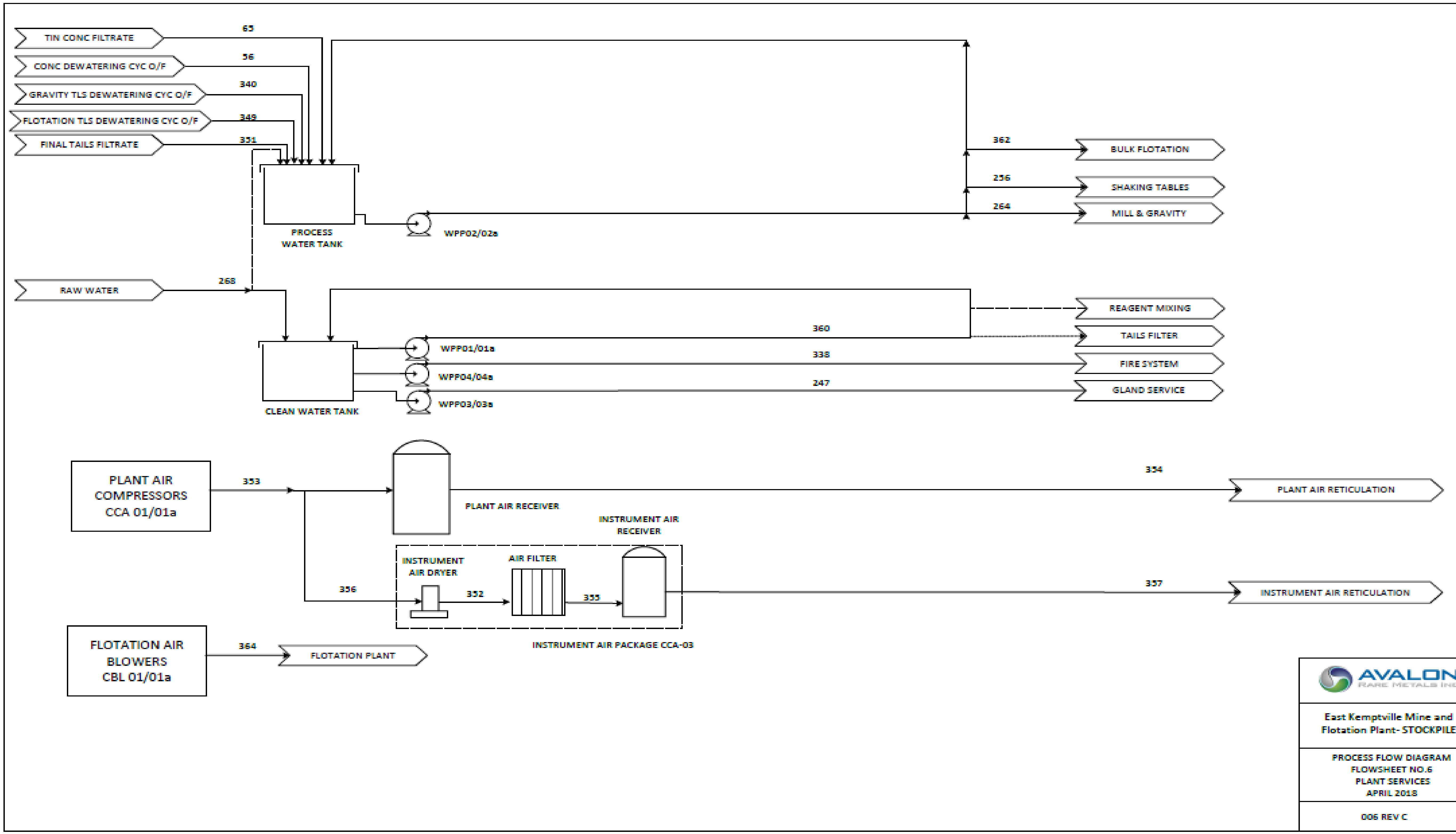


Figure 17.8
Flowsheets Showing Clean and Process Water

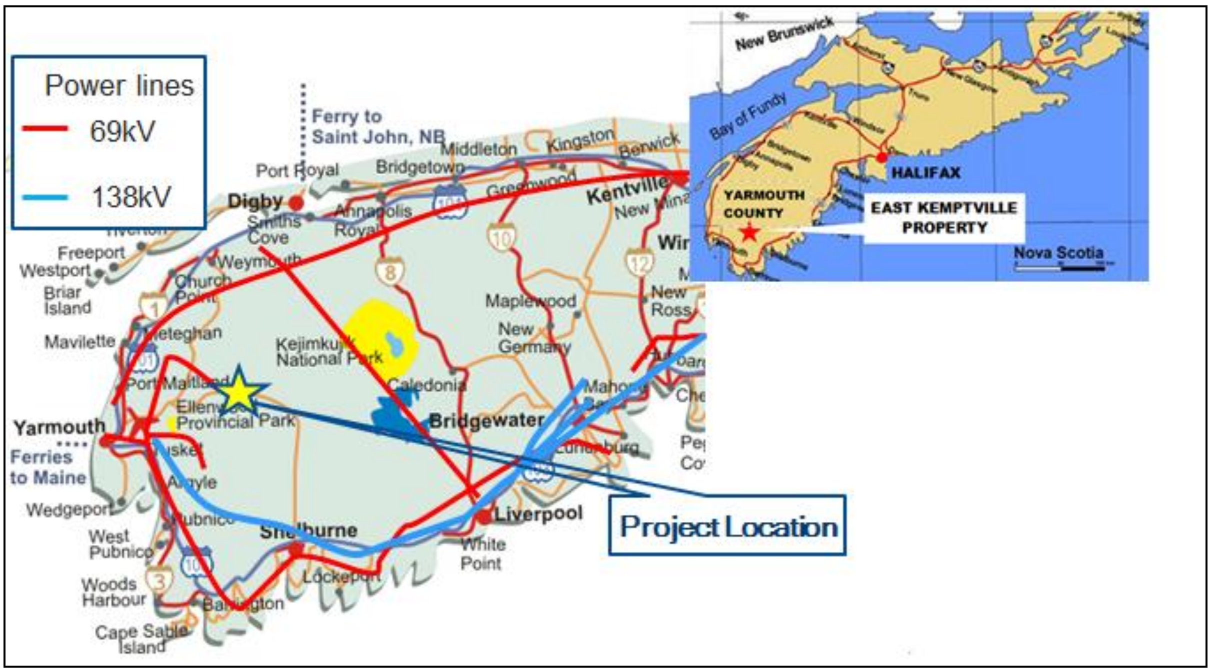


18.0 PROJECT INFRASTRUCTURE

The East Kemptville Project is located in Yarmouth County, in southwestern Nova Scotia. The property is situated approximately 180 km southwest of Halifax, the provincial capital, and 55 km northeast of the town of Yarmouth, a port town of 7,500 people.

Figure 18.1 shows the mine site in relation to major towns, roads and power lines.

Figure 18.1
Mine Location Map



18.1 ACCESS

Highway 203 was constructed in the early 1980s to help facilitate the transportation of concentrates from the East Kemptville mine to the port of Shelburne. Connecting with Highway 340, this highway links the Project site with the port towns of Yarmouth and Shelburne. Shelburne is located on the Atlantic Ocean some 45 km to the southeast of the Project site. The highway is currently in reasonable condition for about 30 km from the junction with Highway 101 but does require some minor upgrading from there to the mine entrance. It is anticipated that this upgrade will be carried out by the province of Nova Scotia.

An existing air strip near Yarmouth can be used for access from outside the area. The airport has two asphalt-surface runways, one 5,000 ft (1,525 m) long, the other 6,000 ft (1,830 m) long, and is capable of handling B737/A320 equivalent aircraft. Both Avgas and Jet A-1 fuels are available. There are four RNAV GNSS instrument approaches, VOR (VHF omnidirectional range/distance measuring equipment) and NDB (non-directional beacon) and a high-intensity approach lighting system.

Existing roads on site allow easy access to the entire site for operations and maintenance. A new haul road will be required from the mine pits to the processing plant, but this distance will be short.

18.2 SITE FACILITIES

18.2.1 Power, Fuel and Water

Primary power to the site will be provided by Nova Scotia Power. A 69-kV line is already in place to the mine site to feed a new substation providing sufficient power for the mine operations. A power distribution system will be established to feed power to the site as required. Emergency/back-up power will be provided by a diesel generator.

Diesel storage and fueling stations will be provided on site for mobile equipment.

Raw water will be taken from the Tuskett River and stored in a tank at site to supply potable water, fire water and process water top-up (process water will be recycled to keep make-up water requirements to a minimum). Excess water will be treated prior to release to the environment.

Process water will also be extracted from the two existing pits or recycled from the TMF in order to minimize raw water consumption and to also make the pits accessible for mining.

A water treatment facility to treat run-off from the tailings is already in operation at the site, consisting primarily of settlement ponds and lime treatment. This facility will be maintained for ongoing operations and modified as required to meet the new project demands, though minimal changes are anticipated to be required.

18.2.2 Buildings, Communication and Waste Handling

The intention is to erect a single pre-engineered and pre-fabricated building that can house the main processing plant (excluding crushing circuit), stores and workshop areas all under a single roof. Containerized/pre-fabricated administration offices, first aid/medical room, a site laboratory and dry facilities will be provided, including lockers and change facilities for both male and female workers.

Mine vehicle/mobile equipment maintenance facilities will be supplied by the mining contractor.

Proven, reliable and state-of-the-art telecommunications systems will be provided at the site for permanent operations and maintenance.

Waste materials (organic waste, hazardous and recyclable wastes, etc.) will be sorted on site and disposed of off-site using local contracting companies or existing municipal handling facilities.

There is sufficient infrastructure in the area to support the labour force required for the Project operations, there will be no requirement for camp accommodations at site.

18.3 CONCENTRATE STORAGE AND SHIPPING

Concentrate will be bagged, containerized and stored at site before shipment on a regular basis to the laydown area at the port in Shelburne or Halifax. On average, approximately 120 t of concentrate will be produced per month requiring the transportation of 4-5 truckloads per month from the site to the port.

The dock at the port in Shelburne has a proven history of accommodating ships and ocean-going barges. The dock is 20 m by 169 m with the ability to accommodate ships up to 190 m in length. Water depths range from 9 m to 10 m at low tide. The dock is designed for axle loads of approximately 10 t which will adequately accommodate the anticipated truckloads.

The port is regularly used to handle containers. Typical container ships docking at the port are around 7,000 t and 130 m in length.

Near the dock is a 75 m by 150 m laydown area that could also be used to store concentrate containers awaiting shipment. While the port is currently only handling approximately 200 containers per year, in prior years, the port has been able to process up to 150 containers per month, more than what is anticipated for the Project. Facilities at Shelburne are illustrated in Figure 18.2.

Figure 18.2
Aerial View of Ship-loading Facilities at Shelburne

19.0 MARKET STUDIES AND CONTRACT

For the purposes of this report, Avalon has undertaken an in-house analysis of the markets for tin concentrates during the course of which it has consulted with industry participants and specialist consultants. Avalon is a member of the International Tin Association (ITA) (formerly the International Tin Research Institute (ITRI)) and has access to all the market information that ITA provides to its members. While the current development model does not contemplate recovery of copper and zinc/indium concentrates, these are potentially recoverable in the future, and are included in some alternative development models. Recent work has demonstrated that the zinc concentrate from East Kemptville would be enriched in indium, a potentially valuable rare metal by-product.

In April 2018 ITA released its Central Forecast for tin in 2022 of USD25,000/t (Source: "Tin Market Outlook" ITA presentation in Budapest April 2018).

The World Bank tin price forecast for 2020 is USD21,038/t and for 2025 is USD20,169/t (Source: Commodity Markets Outlook, World Bank Group, April 2018).

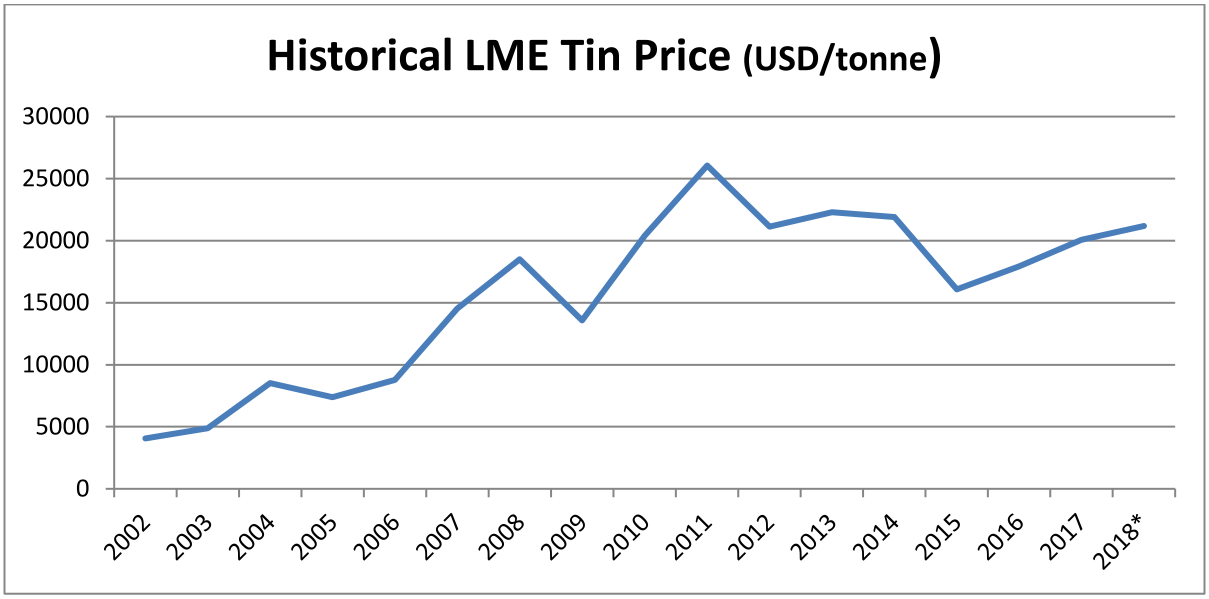
For the purposes of this report, the price of USD21,038 /tonne has been used which is the World Bank forecast for 2020 but also is consistent with the LME price for tin during the first quarter of 2018 (USD21,187).

19.1 TIN HISTORIC PRICES

Tin prices on the LME were below USD10,000/t from 2000 to 2006 and then increased dramatically in 2007 and 2008 to USD20,000/t before falling back to USD10,000/t in mid-2009. Since 2009 the price has fluctuated in an upward direction and maintained an average above USD21,000/t (See Figure 19.1). The low prices prior to 2008 did not encourage much tin exploration or mine development, and price volatility has not encouraged significant tin exploration since. As a result, there are relatively few new tin mines or projects that have reached advanced stages of development in the developed world. The exception is in Myanmar where ITA has reported on a surge of artisanal tin production in recent years with concentrates being shipped to China for processing. It is unclear if this production is sustainable over the longer term.

Prices in 2017 traded between USD19,000 and USD21,000 with an average of USD20,061. In 2018 through March the price has averaged USD21,187/t on the LME.

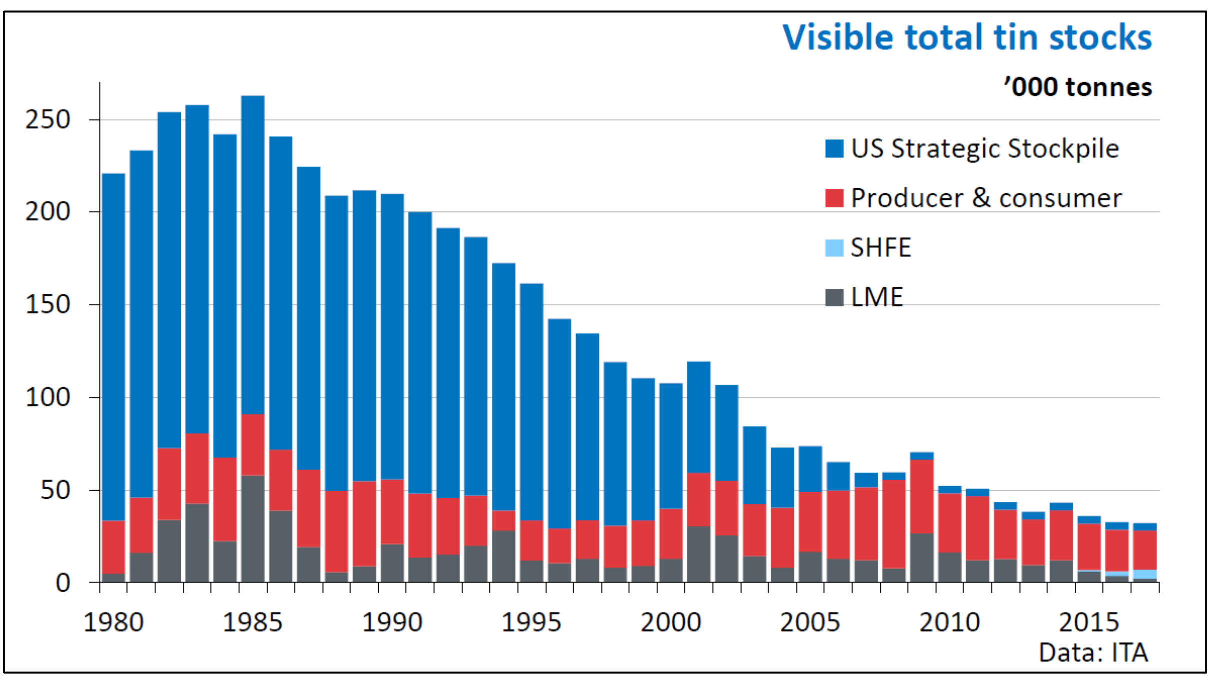
Figure 19.1
LME Average Annual Tin Prices



*2018 data is for the period January to March 2018 (incl).
Source: World Bank Commodity Price Data (The Pink Sheet), and Commodity Markets Outlook April 2018.

According to ITA, world stocks were at their lowest level since the 1980s (Figure 19.2) meaning that any shortage of production should have an immediate positive impact on prices

Figure 19.2
Tin Stocks



19.2 TIN CONTRACT TERMS

It is anticipated that the East Kemptville Project will produce a tin concentrate grading 55% tin. Historically, the East Kemptville mine produced a tin concentrate with average assays as shown in Table 19.1.

Table 19.1
East Kemptville Tin Project Historic Tin Concentrate Assay

Element or Oxide	Average (%)	Historic Range (%)	
		Low	High
Sn	50.00	45.0	60.0
Fe	1.69	1.0	2.5
Cu	0.029	0.01	0.04
Pb	0.003	0.001	0.005
Zn	0.036	0.02	0.005
Bi	0.008	0.005	0.01
Cd	0.0003	0.0001	0.0005
Mn	0.003	0.002	0.005
Hg	<0.0001	<0.0001	
Nb_2O_5	0.22	0.10	0.30
Ta_2O_5	0.018	0.01	0.03
Ti	0.13	0.05	0.2
WO_3	1.32	0.5	2.0
U	0.014	0.01	0.02
Th	0.006	0.003	0.008
As	0.003	0.001	0.005
Sb	0.001	0.0008	0.0015
F	5.0	2.0	7.0
S	0.20	0.10	0.25
SiO_2	14.30	10.0	18.0
Al_2O_3	15.10	10.0	18.0
CaO	0.52	0.2	1.0
MgO	0.021	0.01	0.05
K_2O	0.26	0.1	0.5

Tin concentrate terms are typically stated as a percentage of the LME price or deduction from the LME price, multiplied by the tin content in the concentrate, less deductions or bonuses based on the concentrate assay. Typically, a minimum tin content in the concentrate is specified for acceptance, as are maximums for various deleterious elements. Tin smelter contracts are normally based on an amount of payable tin that varies between 85% and 92% of the price and a treatment charge that is normally 10% of the price. Penalties are imposed for deleterious elements above a certain level and these penalties vary by smelter.

Smelting and refining charges fluctuate based on smelter demand for concentrate, concentrate assay, volume of concentrate available and the perceived reliability of the supplier. Avalon has conducted discussions with several potential concentrate buyers and market consultants.

In the case of the East Kemptville Project, containerized concentrate could be shipped out of Shelburne or Halifax, Nova Scotia. Shelburne is the nearer port and has the ability to provide container service on a call basis but does not offer regular container service. For the purposes of the economic analysis presented in this report, established rates from Halifax have been used since it offers regular liner container services. The prime destinations for the concentrate would be smelters in Europe (Belgium, Germany) and in Malaysia or Thailand.

Based on current shipping rates for 20-ft containers and assuming 20 t of concentrate per container, shipping charges are estimated at CAD175/t concentrate to major European ports (Antwerp or Bremen) and CAD225/t concentrate to ports in Thailand or Malaysia. These charges include inland trucking costs from East Kemptville to Halifax.

Avalon has signed an indicative off-take agreement in the form of a non-binding Memorandum of Understanding (MOU) with a well-known, large tin smelting company (who wishes to remain anonymous for the time being) for all of the East Kemptville tin production. Based on this MOU, the calculated price Avalon will receive for its tin concentrate will be USD10,621/dmt (before freight charges of CAD225/t).

19.3 ZINC/INDIUM AND COPPER CONCENTRATES

The current, proposed Project does not make provision for the production of either a zinc or copper concentrate, nor is any recovery of indium contemplated at this time. However, Avalon will continue to monitor prices for these potential metal by-products and should the production of concentrates of any or all become economically viable, the option will be there to enlarge the operation to facilitate their recovery.

Historically the zinc concentrate produced at East Kemptville had high indium content which would be a valuable by-product. Zinc smelters who recover indium are located in Canada (Trail, BC), France, and Korea. Indium is relatively rare in the earth's crust and total world demand was just above 800 tons in 2016. Indium is used as an alloy material such as indium tin oxide (ITO) which is used in flat panel displays, smart windows, electronics and thin film photovoltaics for solar panels. Indium prices have been very volatile in recent years due in part to the failed Chinese Fanya Exchange that helped fuel a price increase, and then a collapse in late 2015 to USD270 per kg, its lowest price in 11 years. Prices in mid-2016 had not yet recovered and were just above USD200/kg, however discussions with market participants in April 2018 revealed that prices had firmed up to a range of USD330-USD350/kg level.

19.4 CONTRACTS

In early 2018, Avalon entered into a non-binding MOU for the sale of all its production of tin concentrate with a well-known company that owns a large tin smelter. The formula used by this customer for determining concentrate pricing has been used by Avalon in the financial model.

Avalon has not entered into any material contracts that are required for property development.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Following the completion of an environmental baseline study, impact assessment and permitting, the East Kemptville mine operated between 1983 and 1992 at a production rate approximately four times higher than that envisioned for the currently proposed Project. After 1992, the mine entered a care and maintenance phase under provincial approval which remains in place. The overall site is currently considered a brownfields site with ongoing management and treatment of surface water.

The East Kemptville site has long-term surface and groundwater environmental liabilities (RAL, East Kemptville 2017 Annual Report, 2018) that are the result of sulphide minerals that remain in the pit walls, low grade and waste rock stockpiles, and tailings, all of which generate acid mine drainage to a greater or lesser extent. At this time, these liabilities are being effectively managed by the surface rights holder through the collection, treatment and release of treated water. Compliance monitoring and performance of effluent treatment is summarized in annual reports by the surface rights holder and submitted to the relevant provincial regulators (RAL, East Kemptville 2017 Annual Report, 2018).

A three-party agreement between Avalon, Rio Algom (RAL, the surface rights holder) and the Government of Nova Scotia will be required, prior to development of this Project, which details how and when Avalon will assume care and custody of the closed site as a precursor to new mining activities. A letter describing this requirement was signed by the Ministry of Natural Resources (now Nova Scotia Energy and Mines) and Ministry of Environment. Other than the long-term liability associated with the mine waste and water treatment requirements noted herein, Avalon is not aware of any instances of fuel oil spills or other sources of significant contamination, or any regulatory orders that have been issued against the site. This information will be confirmed once the agreement, referenced above, has been finalized.

Construction and operation of the proposed Project is not anticipated to be subject to approvals under the current Canadian Environmental Assessment Act 2012 (CEAA) as the mine does not exceed any of the existing CEAA triggers, including mine and mill tonnages and footprint expansion. The pending changes to federal environmental impact assessment legislation are also not anticipated to require approvals from the Project; this will be confirmed when the details of the finalized legislation are available. The Project is not anticipated to have any substantive new impacts to terrestrial, fish or fish habitat, and will not impact any federally designated wildlife conservations areas. The Project will be subject to the Nova Scotia Environment Act and associated regulations (including the Environmental Assessment Regulations), via the provincial "One Window" approach to mineral resource development chaired by Nova Scotia Energy and Mines.

The proposed mine operations are an integral component of the overall mine rehabilitation strategy and to mitigate the present and ongoing sources of environmental liability. The brownfields site has known sources of acid mine drainage (AMD) to both surface and groundwater. These are well understood by Avalon and appropriate mitigations and monitoring for these historical impacts have been developed. An updated closure plan is currently under development and will be a requirement of the Project permitting process.

20.1 ENVIRONMENT AND SOCIOECONOMIC CONDITIONS

20.1.1 Environmental Setting

Following the completion of an environmental baseline study, impact assessment and permitting (East Kemptville Project Environmental Assessment, 1983), the East Kemptville mine operated between 1983 and 1992 at a production rate approximately four times higher than that envisioned for the currently proposed Project. In the early years of operation, a discharge of elevated turbidity from the TMF occurred. Modifications to the management of the TMF were immediately put in place and based on Avalon's current understanding the system has been operated successfully since that time. Upon cessation of mining, in 1992, a closure plan was filed and accepted by the NSDNR, in 1993. All mill facilities were decommissioned and demolished and portions of the Project site have been regraded and revegetated. The tailings surface has been revegetated with on-going maintenance periodically required. Much of the rest of the site has also been revegetated, but full restoration to a natural state has not yet been undertaken. Waste rock stockpiles are sparsely revegetating naturally.

The East Kemptville site has long term surface and ground water environmental liabilities that are the result of sulphide minerals that remain in the pit walls, low grade and waste rock stockpiles, and tailings, all of which generate AMD to a greater or lesser extent. These drainages impact both surface run-off and groundwater. At this time, these liabilities are being effectively managed by the surface rights holder through the collection, treatment and release of treated water (East Kemptville 2017 Annual Report, 2018). Two water treatment plants, seepage collection and pumping systems are currently in operation on the site to manage the impacted waters. Based on informal discussions with the local regulators, no known directives have been issued to the surface rights holder in response to any non-compliance with conditions of the existing Industrial Approval (2009-06752). Extrapolating current water quality trends (East Kemptville 2017 annual report, 2018) suggest that treatment could be required for 40 years or more, especially in consideration of provincial discharge and closure requirements and the application of the new MDMER requirements. It is also anticipated, based on site water treatment experience, that as the water quality prior to treatment slowly improves, it becomes increasingly difficult and expensive to treat the water to acceptable metals concentrations. This now includes requiring the periodic addition of flocculants to facilitate precipitation of metals, primarily zinc, and this use is predicted to increase.

The Tusket River and, more recently, Big Meadow Brook adjacent to the Project and the receiving waters for the treated mine effluent, support recognized sport fisheries. The Tusket River watershed is also known to contain rare species of coastal plain flora, at distance, downstream. Plant species of conservation concern are found in the wetlands, barrens and along lake shores within the watershed.

As part of a comprehensive due diligence exercise, the surface rights owner has provided Avalon with extensive background information of the site. This included the original development Environmental Impact Assessment, details of the treatment systems (their upgrades, procedures, maintenance, monitoring and performance), Dam Safety Reviews and annual Dam Safety Inspection reports, as-built diagrams, historic and recent biological studies, site water and ground water monitoring data and ground water model, annual reports and meeting minutes with local NGOs. This information, where relevant, will contribute to the Registration required for the proposed project under the Nova Scotia Environmental Assessment Regulations.

Water quality is monitored according to the existing Industrial Approval (2009-06752) at identified locations and frequencies. Some additional biological monitoring has voluntarily taken place in the receiving water body by the surface rights owner; this is anticipated to become a requirement, should Avalon re-start operations, due to new mining effluent regulations that have come into effect since the original mine closure. This monitoring includes water and sediment quality, along with an assessment of benthic invertebrate and fish populations (Minnow Environmental Ltd., 2016).

Additional aquatic analysis was also completed to assess potential long-term risk for parameters presently under review for changes to the Federal Metal and Diamond Mining Effluent (MDMER) Regulations, which will apply if operation re-starts. An assessment of potential optional and cost-effective treatment technologies is ongoing to address the anticipated application of more stringent effluent standards. The existing TMF was originally designed, optimized (once being used) and successfully operated for plant and water volumes and associated reagent use much larger than the production rate proposed by this Project (East Kemptville Project Environmental Assessment, 1983).

20.1.2 Socio-economic Setting

The East Kemptville site is located 55 km by road east of the town of Yarmouth, Nova Scotia, and accessed from Yarmouth via Nova Scotia paved highways 340 to Carleton and then 203 to the site. Yarmouth is on Highway 103 and Highway 101, 300 km by road from Halifax. The town of Yarmouth has basic requirements for operations and would be, along with Shelburne, the likely location for employees to reside, as well as the rural areas and small communities in between. Yarmouth had its highest population, in 1961, at 8,636 and at present is believed to be of the order of 6,700. The population has been dropping steadily in recent years due to lack of economic opportunity.

The main economic activity of the community is lobster fishing although the Nova Scotia government is attempting to promote tourism. There are no communities in the immediate vicinity of the mine. In a direct line, one tourist operator is located approximately four km from the site, as is the community of East Kemptville.

Avalon has initiated engagement with the local community. It has hired a local geologist and drilling company to assist in supervising the initial drilling program. It has held positive meetings with the Tusket River Environmental Protection Association, the Yarmouth Mayor and key members of her team, and the Chambers of Commerce of Yarmouth and Shelburne, with plans to expand this engagement in the future to the Municipality of Argyle and others. Members of both provincial and federal government have also been regularly updated on the Project and are also supportive. A recent meeting and site visit, on July 19, 2018, included representatives from the Nova Scotia Department of Natural Resources, Department of Environment, the Acadia First Nation Band along with Avalon and RAL. Given the depressed level of industrial activity and following these preliminary discussions, community support for the Project is anticipated. Ongoing engagement with other stakeholder groups is also planned.

The Nova Scotia mining industry in this region has undergone a loss of employment in recent years. In addition, employment in many other areas of the Nova Scotia economy has also seen reductions. As such, it is anticipated that there will be an available educated workforce for both construction and operations at the site, composed of both Indigenous and other local employees.

20.1.3 Indigenous Communities

Avalon is recognized for its leadership in Indigenous Engagement and the province of Nova Scotia is committed to the clear and transparent process for engaging Indigenous government/communities. Avalon has already reached out to representatives of the Mi'kmaq First Nation, specifically the Acadia First Nation Band, to inform them of recent small drill programs and to initiate dialog. Recent discussions have included a Project overview, a site visit and discussion on both operating and closure. Avalon also helped a local Mi’kmaq business development organization to develop a core box construction business and supply core boxes to the Avalon drill program.

20.2 ENVIRONMENTAL REGULATORY REQUIREMENTS

20.2.1 Provincial

The Nova Scotia government supports mineral development through its “One Window” regulatory process to aid proponents with reviewing, permitting and monitoring mine development projects in Nova Scotia. This includes the Nova Scotia Energy and Mines, Nova Scotia Environment, Nova Scotia Department of Labour and other provincial, federal and municipal agencies as determined on a project-by-project basis. Avalon has initiated development of relationships with the regulators through participation in a multi-ministry meeting, though this was based on a previous business model. Since then, the various applicable regulatory agencies have been periodically updated in association with the development of the current business model. A multi ministry meeting and site visit with key provincial regulators and representatives of the Mi'kmaq First Nation was held most recently in July 2018.

A three-party agreement between Avalon, RAL and the Government of Nova Scotia will be required to determine Avalon's responsibilities for the on-going care of the closed site, prior to development of the currently proposed Project. A Mining Lease and Crown Land Transfer under the authority of the Nova Scotia Energy and Mines is also required. These have been initiated, and the final application is planned for September, 2018.

Avalon has been notified that a Registration under the Nova Scotia Environmental Assessment Regulations is required for the proposed Project. It is anticipated, that a new Industrial Approval (under Activities Designation Regulation) will also be required. Based on preliminary discussions with regulators, it is expected that the Registration will be able to rely on information available from previous and ongoing studies and monitoring reports where applicable and will be supplemented with new information from limited field programs. Other provincial environmental regulations that govern water quality, waste management, fuel storage, air quality, wetlands and watercourses, among others, would also apply. Discussions have been held with relevant provincial regulators toward streamlining the regulatory approval process.

In addition to, ongoing surface and groundwater water quality monitoring and biological aquatic effects assessment, long term leachate studies and detailed acute and sub-lethal aquatic toxicity studies are ongoing to further assess and validate that the effluent treatment system will continue to meet regulatory chemical and biological requirements (East Kemptville 2017, Annual Report, 2018). A Species at Risk (SAR) study has been initiated for breeding birds (complete), vegetation (September, 2018) as well as a wetland survey (September, 2018). Risks to affect SAR species are considered low due to the disturbed nature of the site and small footprint of new Project activities. This information will also be used as components of Project Registration and provincial permit applications, for example wetland alteration permits, if required.

20.2.2 Federal

The new Project is not anticipated to be subject to approvals under the current Canadian Environmental Assessment Act 2012 (CEAA) as the mine does not exceed any of the CEAA triggers, including mine and mill tonnages and increase in mine footprint. The pending changes to federal environmental impact assessment legislation are also not anticipated to require approvals from the Project; this will be confirmed when the details of the finalized legislation are available.

The Project is not anticipated to have any substantive new impacts to fish habitat, regulated by Fisheries and Oceans Canada or other areas of federal environmental regulation (e.g., migratory birds). While there has been an ongoing treated mine water discharge at site since it began operations, effluent treatment studies with pilot plant and site water have been initiated to assess if there would be any anticipated substantive changes to effluent quality and quantity and thus the receiving waters of Big Meadow Brook and the Tusket River. It is anticipated that the new Project will be subject to requirements under the Metal and Diamond Mining Effluent Regulations (Fisheries Act).

20.3 WATER MANAGEMENT

A water management plan exists for the site, and a new plan will be required for new Project activities for the Mining Lease and prior to construction. The quantity and quality details below are based on the mine schedule and associated information used in the production/financial model developed by Avalon and Micon.

The Tusket River and, more recently, Big Meadow Brook, both adjacent to the Project, are recognized sport fisheries. The Tusket River is also known to contain rare species of coastal plain flora at a distance downstream. Nationally endangered and threatened plants are found in wetlands, barrens and along the lake shores in the watershed. There is some merit in the present and potential future discharge providing a benefit to Big Meadow Brook and the Tusket River in terms of fisheries habitat, by the neutralization of naturally occurring organic acids that are generated upstream of the site by alkaline effluent from the Project.

Due to the small scale of the Project and plan to recycle water from the TMF, water volumes from the TMF during operations are not anticipated to significantly change from present discharges. During construction and the first year of operation, there will be an average flow increase of approximately 35% during the period of pit dewatering, but these will remain below historical peak discharge rates. Pit water will be pre-treated, in-pit, to ensure adequate water quality prior to pumping for direct discharge to the receiver, thus not changing the retention time and proven treatment capability of the existing TMF system. A preliminary proven design for the removal and storage of sludges at the bottom of the two pits has been costed, though it is believed there is opportunity to improve on this.

Water quality requiring treatment is also not anticipated to vary significantly from historical operating concentrations, and operating volumes will be much smaller than the original Project. Unlike the historical operations, virtually no reagent is planned in the early stages. Gravity separation of tin only and potentially a small sulphide float to clean up the concentrate (using a degradable flotation reagent and 100% recycle of this water) is planned, such that no impacts to the operation of the existing proven effective water treatment facility or the downstream receivers are envisioned. In conclusion, there will be no changes to the water treatment systems, effluent quality and quantity or air emissions, and also no change in the Project footprint (i.e., no new disturbed areas).

Planned operations are an integral component of the rehabilitation strategy and to mitigate the present and ongoing sources of environmental liability. Tailings will be discharged and deposited sub-aqueously, starting in the existing TMF in Years 1-4, to prevent oxidation and acid mine drainage in perpetuity. In late Year 4, tailings will start being deposited in the mined-out Baby Pit. Beginning in Year 6, an additional flotation step will become operational in the concentrator to produce clean, low sulphur, low permeable tailings to be placed on and become a clean cover for isolating the existing TMF. Thus, in Years 6-8, due to the production of an estimated 500,000 tonnes/year of clean, low sulphur, low permeability tailings from the tailing stream, significantly smaller quantities of sulphide bearing tailings will be sent to the Baby Pit. The Baby Pit will be full to within approximately one metre of the water surface early in Year 8. In Years 8-11, similar quantities of tailing will be placed into the Main Zone Pit South. At closure, they too will be permanently stored under water. The clean tailings will be placed as a cover on the existing, coarse tailings pile in the TMF from Years 6-11. The minimum one-metre thick cover is designed to isolate the TMF from water inflow (and to a lesser extent oxygen), thus eliminating the TMF as a source of Acid Mine Drainage.

Process water discharged to the Baby Pit or Main Zone Pit will subsequently flow into the TMF, thereby increasing water retention times and treatment efficiency during the periods of additional flotation. The cover will be completed in Year 11. The cover will be progressively remediated as it is deposited to prevent erosion. Monitoring and maintenance as required will ensure a self-sustaining vegetated cover by the time the Project is closed in Year 19, as well as validate the effectiveness of the strategy to prevent the release of acid mine drainage.

During Year 8, tailing will begin to be deposited in the Main Zone Pit and for the remainder of the life of the Project. Once dewatered, water will initially be pumped from the Main Zone Pit to the TMF until the end of mining in Year 16, after which time the pit will be allowed to flood and gravity flow to the TMF once it is full. Revegetation in the TMF ponds can begin in Year 5 and in the Baby Pit in Year 10. Thus, most of the site will be progressively rehabilitated and there is significant time available to monitor the success of the overall rehabilitation program and to fine tune it during operations if necessary, significantly reducing the rehabilitation risk.

All waste rock for the purposes of this study is assumed to be acid generating. The rock from the Baby Pit and the Main Zone Pit will be placed in the Main Zone Pit, either in areas not planned for mining or in mined out areas as the Project progresses. The tailing produced in Years 8-16 will also remain in the Main Zone Pit. All waste rock will remain underwater upon cessation of mining and flooding of the Main Zone Pit. Tailings not utilized for the tailing cover will be placed in the Main Zone Pit in Years 10 and 11. Given that all tailings in Years 12-19 will be placed in the Main Zone Pit, flooding is expected to be completed well in advance of mine closure in Year 19.

The Main Zone Pit has sufficient space for the remaining tailings and all the new waste rock produced, which as currently designed will approach an elevation of 87.5 masl. Therefore, approximately 0.9 Mm^3 of additional capacity is still available at the end of the Project should additional resources be developed elsewhere. Alternately, if necessary due to ongoing generation of AMD from historical waste rock dumps, some of this excess capacity could be utilized for placing these dumps underwater or being processed and placed underwater in the Main Zone Pit. As such, all the potentially acid generating waste rock and tailings will remain below the water surface and are eliminated as a source of AMD. In Years 15-19, the remainder of the low-grade legacy stockpile will be processed until closure during Year 19. No waste rock will be generated during this period, and all tailings will be placed sub-aqueously in the Main Zone Pit. Thus, the low-grade stockpile is eliminated as a source of AMD.

Studies have indicated that pit water from surface to bottom in both pits is already of a quality close to or meeting effluent discharge criteria. A small amount of in-pit treatment will be completed as necessary utilizing excess lime capacity in the existing treatment plants. Most of the water is planned to be discharged directly to the receiving streams at or near the final effluent once acceptable water quality is reached. Pumping will be initiated from the Baby Pit during construction at a rate of approximately 230 m^3/hour. This represents a 30% increase relative to the last 7-year annual average discharge rate from the TMF, and marginally higher than the maximum annual average rate in that time period and can be managed to remain below peak discharge rates. However, the maximum pump rate will be controlled to below historical maximum water discharge rates. This relatively small and short-term increase in discharge rate is not anticipated to further impact the downstream fish and fish habitat and is believed to be well below the discharge rates of the former operations that operated at a tonnage rate of approximately 4 times the planned Project rate.

Once the Baby Pit is pumped out, pumping will begin in the Main Zone Pit. The total pumping time is estimated to take approximately 2.5 years and is timed to meet the mining needs and minimize additional flows to the receivers. This strategy will also maximize retention time for the treatment of operational water in the existing TMF. An assessment of this discharge on Big Meadow Brook and the Tusket River, as well as engaging with communities of interest and regulators, is planned to select the preferred point of discharge to ensure that no impacts result from pit dewatering. Both pits could be utilized to supply process water to the concentrator to reduce both fresh water inputs and reduce the overall discharge volumes to the receivers, but recycling from the TMF is the preferred option to maximize retention time for process water treatment in the TMF. It is also planned to assess the quantity of sludge presently in the TMF Sludge Pond, and pending the results, may also have to be dredged and disposed of in a similar manner to the sludges in the two pits.

No new areas with fisheries or fish habitat will be utilized and further impacts to the freshwater aquatic environment are not anticipated. Most process water will be recycled from the TMF to minimize fresh water use and discharge volumes, and only a small volume of fresh water from the large Tusket River will be required for the small operation. This water will be treated and largely used for potable water purposes. There is an existing road and power line to the site, such that approvals for these will not be required, except perhaps for some upgrading. No camp is planned for construction or operations. Limited sewage treatment is proposed using septic tanks and field bed. There will be no landfills established on site during operations. All waste other than waste rock and tailings will be sorted and temporarily stored on site prior to being sent to appropriately licenced waste management facilities.

The brownfields site has known sources of acid mine drainage (AMD) to both surface and groundwater. These are now well understood by Avalon and appropriate mitigations and closure plans identified for these historical impacts have been developed, as well as any impacts anticipated from future operations.

The surface rights holder has completed a number of studies in recent years, including a Tailing and Water Management Optimization Study, a Hydrological Study, and a Geotechnical Investigation of Tailings Dam and Coarse Tailings Pile, completed by independent consultants who are expert in these areas. Annual Dam Safety Inspections and periodic Dam Safety Reviews have been completed. These reports indicate that the existing tailings facility stability is acceptable with ongoing management, and no remediation is required. The tailings and water optimization study identified a number of upgrades to improve water treatment efficiency and reliability which have subsequently been implemented.

Avalon has evaluated the existing Tailings Management Facility (TMF) for use in future operations from both a long-term facility safety/stability and tailings pond capacity perspectives. Additional drilling has been completed in the area of the north facility to assess long term stability as well as to measure tailings metal values for potential re-processing. (Tailings reprocessing is not envisioned in this project model.) The additional study indicated that the TMF is stable and identified additional mitigation measures will not be required. Avalon proposes to utilize the pond sections only of the existing facility for sulphide bearing tailings, and only in the first 4 years of operation. As the new tailings are anticipated to be acid generating, all the sulphide bearing tailings will be placed a minimum of one metre below the existing water surface to prevent oxidation and the formation of AMD. The one metre cover to be placed on the existing tailing pile will have the additional benefit of lowering the water table in the tailing pile, further improving TMF stability.

An assessment of the quantity and quality of the treated metal bearing sludge presently located in the bottom of the two pits has been completed. A complex treatment strategy to manage this material has been developed and costed. Opportunities to optimize this strategy will be more fully assessed during the Feasibility Study. For example, the material in the Main Zone Pit could be pumped into the Baby Pit.

20.4 MINE CLOSURE, WASTE MANAGEMENT AND PROGRESSIVE REHABILITATION

A mine closure plan exists for the site, but a new plan will be required for new Project activities for the Mining Lease and prior to construction. The quantity of tailings and waste rock produced as detailed below is based on the mine schedule used in the production/financial model developed by Avalon and Micon. This Project's primary objective is to economically reduce or eliminate the long-term environmental liability of the site. This includes the elimination of the existing perpetual treatment strategy by permanently stabilizing the site such that ongoing treatment of AMD will no longer be required. The strategy's objective is to eliminate the water treatment risk and also to reduce the already low physical risk associated with the facilities. By implementing a low permeability cover on the existing Coarse Tailings Pile (CTP), water levels in the CTP would remain at permanently low elevations and thus lower the risk of CTP failure. The extensive progressive remediation built into the development model and identified below is anticipated to enable some of the financial assurance to be returned during the life of the Project.

The strategy for a "walk away" closure involves progressively isolating all potentially acid generating waste rock and tailings from oxygen and/or water. This either stops oxidation and the formation of AMD or prevents the flushing of oxidized materials into the environment. The strategy proposed includes:

1. Processing of the low-grade stockpile and placing the resulting tailings a minimum of one metre below water in the TMF and the pits. This reduces oxygen exposure to very low concentrations and reduces the production of AMD to acceptable levels. This eliminates the coarse stockpile as a major source of AMD. If required, waste organic materials may be added once deposition is completed to aid natural regeneration of wetland vegetation to further stabilize the sulphides by aiding in the generation of anaerobic reducing conditions. In the unlikely event of a small quantity of sulphide remaining, the goal would be to naturally treat this by creating wetland conditions.
2. Isolation of the existing tailings management facility CTP with a low permeability clean tailings cover. This would reduce water and oxygen penetration and eliminate the flushing of historically oxidized products into the environment. The cover would then be progressively revegetated for stability, aesthetics and beneficial reuse by biota during operations. Beneficial reuse by stakeholders for agricultural production is also being discussed.
3. Placement of all waste rock produced during mining of both pits into the Baby and Main Zone Pits where it will be flooded on closure. Again, the objective would be to prevent the oxidation of sulphides and the generation of AMD.
4. Assess the North and South Waste Dumps to determine their contribution to the site AMD. Given that they both contain relatively low-grade materials and they have been oxidizing for many years, they are less likely to be significant sources of AMD at closure. Should this assumption be determined to be incorrect, remaining acid generating materials could also be relocated at minimal cost into the Main Zone Pit, either through direct trucking or through processing, whichever is more economic.

The Baby Pit will be dewatered during construction. The Main Zone Pit will be dewatered by the end of Year 1 or very early in Year 2. The first year of operation will only utilize material from the Low-Grade Stockpile and tailing will go underwater in the TMF ponds. The mining of the Baby Pit is planned in Years 2 and 3 which has the additional benefit of generating space for the tailings deposition beginning in late Year 4. During the mining of the Baby Pit, the tailings will also be placed in the TMF ponds. Mining of the Main Zone Pit begins mid-Year 3 until Year 14. During Years 6-11, a clean tailings cover will be used to cover and isolate the existing CTP. During Years 7-12, the cover will be progressively revegetated, and the impact of the clean cover can be monitored to demonstrate the effectiveness of the cover and vegetation to isolate the pile and prevent release of acid mine drainage. Tailing will begin being placed in the Main Zone Pit when the Baby Pit tailing reaches one metre from the water surface in late Year 8. The low-grade stockpile will be processed during the full life of the Project and the tailings placed underwater in the mined-out pits and TMF.

In order to develop the waste rock and tailings management strategy for the Project, an assessment of the new waste rock and tailings volumes and the existing volumes presently available in the TMF, Baby Pit and Main Zone Pit was completed. The key objectives were to:

1. Confirm that there was sufficient space to place all future potentially acid generating waste rock and all acid generating tailings under water with a minimum 1 metre water cover to prevent oxidations, with an objective to allow a "walk away" closure plan for these materials.
2. Minimize waste rock and tailings management costs.
3. Continue to utilize the existing proven TMF for all mine tailings and surface runoff water treatment during operations.

Hydrometallurgical testing was completed and demonstrated that sufficiently low sulphide "clean" tailings can be produced with the necessary physical characteristics to act as a low permeability cover for the CTP.

In an effort to be conservative, all waste rock and the non-desulphurized tailings were assumed to be acid generating. Conservative estimates of the density of waste rock and settled tailings were utilized. It was assumed that all tailings would be placed underwater except when the clean tailings cover material is produced from mill feed sourced from the Main Zone Pit mineralization. Approximately 1.7 Mm^3 (2.7 Mt) of desulphurized low permeability tailings (at less than 0.1% S and $<1x10^{-4}$ cm/s permeability) will be placed on the TMF. This is generated from approximately 65%, or 500,000 t/y of the mill feed shipped in Years 6-11 that will be processed through the additional flotation circuit to produce a low-sulphur, low permeability cover that will isolate the existing tailings management facility as part of the closure plan. Since the existing low-grade stockpile sulphides are significantly oxidized, efficient sulphide removal from the stockpile material is anticipated to be problematic, such that fresh mineralization is required to produce this cover. For this reason, un-oxidized mineralization from the Main Zone Pit is proposed for this.

To accomplish this and maintain a sufficiently positive cash flow to achieve a "walk away" scenario, in Year 5 and to confirm that adequate space is available for subaqueous tailing disposal, a flotation circuit will be installed in the concentrator to float the sulphides from the tailings of the un-oxidized Main Zone mineralization. This circuit will produce fine, low permeability clean tailings that can be utilized to cover and isolate the existing TMF coarse tailings pile. The sulphide bearing float will be disposed of under water in the Baby Pit and Main Zone Pit or potentially sold if an interested buyer can be identified. By isolating the existing CTP tailings in this manner, oxidation products that presently contribute to acid mine drainage are permanently contained in the coarse tailings pile. This will eliminate the need for long term acid mine drainage treatment from this source. A 65% recovery of clean tails from the un-oxidized mill feed from the Main Zone Pit will provide the required quantity of "clean" tailings cover during Years 6-11.

The assessment included an analysis of when the various storage locations would become available in comparison to when the materials would be produced. Based on this conservative analysis and schedule proposed, it was determined that there will always be ample underwater storage capacity for all the tailings and waste rock produced. The mine plan sequence was specifically designed to provide safe, adequate access for simultaneous mining and tailings backfill operations by ensuring that tailings fill would remain 1 metre below active haulage areas at all times (See Figures 16.6, 16.7, 16.8). The process for mining and placement of waste materials is as follows:

Year 1: All new tailings from processing of the low-grade stockpile will be placed one metre below the water surface of the existing TMF. There is no waste rock produced in Year 1 as the plant feed comes from the low-grade stockpile.

Years 2 to 9: The waste rock from the Baby Pit and Main Zone Pit will be direct-shipped to four different waste areas located within the Main Zone Pit as discussed in Section 16. All tailings are directed to the TMF in Years 2 to 4.

Late Year 4: When the TMF is filled to one metre below the water surface in late Year 4, all tailings from the Main Zone Pit and Low-Grade Stockpile will then begin to be placed in the Baby Pit (which will be mined out in the current mine plan). This will be sufficient until early in Year 8 when mining of the deep Main Zone South Pit will be complete.

Year 6 to 11: The production of the clean tailings cover will begin and be placed on the coarse tailing pile of the TMF. Progressive revegetation of the clean tailings cover will be completed in Years 7 to 12. The remaining tailing will go either into the Baby Pit (until Year 8) or Main Zone Pit.

Year 15 to 19: After the end of open pit mining in Year 14, the tailings from the final 5 years of stockpile processing can be placed in the Main Zone Pit. No waste rock would be produced after Year 14. At the end of operations as per this scenario, assuming approximately 1.7 Mm^3 of clean tailings are placed on the TMF, there will still be approximately 0.9 Mm^3 of storage space available in the Main Zone Pit should additional capacity be required for processing of material from other mining areas or utilized to relocate any other waste dumps that continue to generate AMD at that time.

A more detailed mine and waste storage area plan, reflective of the envisaged post-closure uses currently under discussion with stakeholders and regulators, will be prepared during the detailed engineering phase of the Project.

20.4.1 Progressive Rehabilitation

Once the TMF ponds are full after Year 4, field studies to initiate the development of wetland habitat on them will be initiated. Depending on water quality and runoff from the coarse tailing pile, this wetland may begin to be progressively established immediately. However, it may be required to wait until some or all of the CTP cover is in place in Year 11 that will stop AMD entering this area from the CTP. Note that the generation of the wetland habitat is not required for long term tailing stabilization, but is done for environmental benefit.

At the point where processing is complete, the Baby Pit will have been flooded for approximately 10 years. Over this time period and starting in about Year 9, utilizing the high quality, high pH treatment plant discharge water, any latent AMD is anticipated to have been treated in situ, and stable good water quality is anticipated to have been achieved. As with the TMF, adequate time will be available to progressively establish, monitor and fine tune a self-sustaining wetland ecosystem utilizing locally available waste organic materials. The objective of this additional level of protection and habitat generation would be to maintain anoxic sediments, prevent tailings oxidation and remove metals from solution, but is not required for long term tailing stabilization. In consultation with the government and local communities of interest, cooperative efforts to establish the endangered Blandings Turtle, known to be present in the nearby Tobeatic Wilderness Area, will be considered for the Baby Pit. Similarly, moose may also be able to utilize this habitat.

At Year 12, the historic tailings will also have been covered and progressively revegetated. Monitoring of groundwater and runoff over the life of the mine and especially following the reclamation of the TMF will measure the effectiveness of the strategy and allow for any required fine tuning well in advance of final closure in Year 20. Runoff and associated AMD from the CTP is anticipated to have been eliminated by this time.

During Years 15-19, in addition to the water associated with the final tailing deposition into the Main Zone Pit, treated water can be diverted from the TMF to aid in flooding the Main Zone Pit with alkaline water to stabilize any remaining oxidized material in the Pit. A thin layer of organic material can be placed on areas that will be sufficiently shallow to support and maintain wetland vegetation to accelerate and aid in maintaining water quality stabilization. Monitoring of the TMF runoff and Main Zone Pit water quality will continue until clean water quality has have been permanently established, with lessons learned from the stabilization of the Baby Pit and TMF over the previous decade or more. The flooding and establishment of stable acceptable pit water quality from the Main Zone Pit is estimated to take 1-3 years following the end of production. Seepage water collection, treatment and monitoring will continue during this period if required and until such time as the cover has demonstrated that seepage and run-off have improved to acceptable water quality. If there are discharges to the environment from the TMF, monitoring of this will also continue although at least some of this treated water in the early years is anticipated to be utilized for the flooding of the Main Zone Pit.

It is noted that discussions with local stakeholders has indicated an interest in utilizing the TMF for beneficial agricultural uses, as has been done in other TMF's in Canada, or potentially for green energy production with solar panels. These discussions will continue. Avalon is also in discussions with local universities to discuss potential crops. Other mines have successfully produced corn, hemp and canola utilizing waste organic covers on TMF's. These will be considered as well as others that may be recommended by Nova Scotia agricultural or traditional knowledge experts.

While engagement on the closure plan will have been ongoing, at this point, if there is a beneficial reuse for any of the buildings or infrastructure on site, in agreement with the new owner and the government, a plan will be developed to transfer the responsibility for that infrastructure to the new owner. This could include maintenance, office or warehouse facilities or agricultural production for example. To the extent practical, remaining equipment and infrastructure will be dismantled and sold for reuse or recycling as scrap. Concrete walls will be knocked down and placed in the Main Zone Pit to add additional alkalinity and clean cover material. This is not anticipated to be required but is a cost effective and environmentally friendly option for the small quantity of alkaline concrete. All remaining unused infrastructure will be removed. All concrete pads will either be placed in the pit or be broken up and covered with organic material and revegetated. At this point and once the pit water qualities are stable, the bulk of the site may be returned to the Crown.

If financial assurances have not been previously returned due to the progressive remediation identified above, as no ongoing water collection and treatment will be required, and the site will be revegetated, most of the financial assurance can then be returned. During operations, opportunities to permanently stabilize the tailings facility dam slopes to eliminate the need for their long-term care and maintenance will be evaluated and implemented, such as the use of available barren pit waste rock for slope protection and stabilization. Once implemented, and the tailings cover performance is proven and effluent quality is acceptable, the site will be in a permanently stable configuration and the full financial assurance can be returned.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

The basis for the PEA capital cost estimate is a processing facility and related infrastructure with a nominal throughput rate of 806,000 t/y of mineralized material, comprising either reclaimed Low Grade Stockpile material or higher-grade mineralization from the 2 pits (Baby Zone pit and the Main Zone pit).

Foreign exchange rates used as a basis for the estimate are:

USD1 = CAD1.30.

Euro 1= CAD1.46.

GBP 1 = CAD1.693.

The estimated Project capital requirements are summarized in Table 21.1 with a more detailed breakdown presented in Table 21.2. All costs are reported as Canadian Dollars (CAD). It should be noted that, apart from the sulphide removal circuit in Year 5, provisions for what might normally be designated as “sustaining capital” are included in the operating costs.

Table 21.1
Initial Capital Cost Estimate

Area	Capex CAD x 1,000	
	Initial Plant	Sulphide Removal (Year 5)
Mining	0	0
Concentrator	18,472	4,076
Tailings Disposal	544	0
Infrastructure	946	0
Total Direct Costs	**19,962**	**4,076**
EPCM	1,497	306
Freight & Transportation	861	188
Other Indirects	1,778	446
Total Indirect Costs	**4,136**	**940**
Owners Costs	1,000	500
Buildings & Tailings	750	100
Contingency	4,820	1,003
Total Capital Costs	**30,688**	**6,620**

Mining capital costs are assumed to be zero as the operation will engage a contract miner and all mining related capital costs are built into the contract mining operating costs. There is no capital provision for the supply and mobilization of mining equipment as it has been assumed by Avalon that a mining contractor will supply all necessary equipment and up-front funding requirements.

The crusher plant has been sized at double the capacity of the concentrator to facilitate a single 12-hour shift for crushing.

The Project intends making use of the existing tailings facility for tailings disposal in the early years of operation, followed by back-filling of the pits once mining operations have ceased.

Excluded from the pre-production capital cost estimate is the allowance for dewatering the two pits. This amount is estimated at CAD850,000, which increases the estimate to CAD31.5 million.

The concept of having most of the plant pre-assembled off-site and delivered in modules (fully or partly assembled) has been assumed for much of the equipment and facilities in order to reduce on-site construction activities.

21.1.1 Concentrator Direct Capital Costs

Based on the results of the various testwork programs, process flowsheets and mass balances were generated together with a detailed equipment list, and process design criteria. From this information, preliminary equipment duties have been determined and budget prices received from qualified vendors. For some of the smaller items, Avalon has used costs from other studies with a similar size or type of equipment.

The pricing of the crushing plant is based on a modular type facility as used in many quarry-type operations.

Site and plant maintenance costs and a provision for site closure are included in operating expenses so no sustaining capital is indicated.

Factors for each area of the processing facility were applied to estimate the associated direct and indirect costs for civil and earthworks, concrete, structural steel, plate-work, piping and electrical/instrumentation. These factors are based on in-house expertise and other similar sized Projects.

Table 21.2
Detailed Breakdown of Initial Capital Cost Requirements

		Mechanical Equipment		Sub	Civil &	Structural	Platework	Piping	Electrical &	Total
		Equipment	Installation	Total	Concrete	Steel			Instrumentation	CAD
DIRECTS										
Crushing Circuit	**%**		5%		7.5%	2.5%	2%	1%	7.5%	
	CAD $	3,210,116	160,506	3,370,622	252,797	84,266	67,412	33,706	252,797	4,061,599
Grinding Circuit	**%**		10%		15%	10%	5%	15%	10%	
	CAD $	5,188,661	518,866	5,707,527	856,129	570,753	285,376	856,129	570,753	8,846,667
Gravity Circuits	**%**		10%		10%	7.5%	5%	7.5%	10%	
	CAD $	3,125,938	312,594	3,438,532	343,853	257,890	171,927	257,890	343,853	4,813,944
Concentrate Handling	**%**		10%		15%	10%	12.5%	12.5%	30%	
	CAD $	378,439	37,844	416,283	62,442	41,628	52,035	52,035	124,885	749,310
Tailings Disposal	**%**		10%		15%	10%	20%	50%	25%	
	CAD $	224,908	22,491	247,398	37,110	24,740	49,480	123,699	61,850	544,276
Infrastructure	**%**		15%		15%	10%	25%	33%	25%	
	CAD $	395,646	59,347	454,993	68,249	45,499	113,748	150,148	113,748	946,385
TOTAL PROJECT DIRECTS	**CAD $**	**12,523,707**	**1,111,647**	**13,635,355**	**1,620,580**	**1,024,776**	**739,979**	**1,473,607**	**1,467,885**	**19,962,181**
INDIRECTS										
EPCM	7.5%									1,497,164
Off-site Infrastructure & Mobile Equipment										100,000
Commissioning & Start-up		2.5%								313,093
Vendor Rep's										250,000
First Fill, 3 Months Consumables										50,000
Spare Parts		2.5%								313,093
Freight & Transportation		5.0%				5.0%	5.0%	5.0%	5.0%	861,498
Contractor Indirects		2.0%								250,474
Insurance		1.0%								125,237
Construction Indirects		3.0%								375,711
TOTAL PROJECT INDIRECTS	**CAD$**									**4,136,269**
Contingency	20%									**4,819,690**
Buildings										**750,000**
Owners Costs										**1,000,000**
TOTAL PLANT CAPITAL COSTS										**30,668,141**

21.1.2 Concentrator Indirect Capital Costs

The EPCM cost is estimated at only 7.5% of direct costs as a result of the crusher plant being a modular/mobile facility, and the use of the vendors to pre-assemble the bulk of the mechanical equipment on skids or modular structure; hence much of the engineering and installation costs will be borne by the vendors. In addition, Avalon will have a small team on site to manage construction so minimal input from the EPCM engineer will be required during the construction period. Electrical design will also be completed by a vendor.

The process modules will be installed in a building that will be a pre-engineered and fabricated steel structure with cladding.

The price of the ball mill is inclusive of liners and the initial ball charge (80 t) has been included in “first fill”.

21.1.3 Bulk Sulphide Flotation Circuit

In Year 5 a bulk sulphide flotation plant will be installed at an estimated cost of CAD6.62M (Table 21.1 above). These capital costs have been estimated using the same philosophy and methodology as applied to the main plant.

This facility will include rougher and scavenger flotation circuits, reagent mixing and dosing equipment, blowers, and a sulphide free tailings filter press. The sulphide concentrate will be pumped to an underwater storage facility while the flotation tailings will be transported by truck to the tailings area for capping purposes.

21.1.4 Tailings

The capital estimate includes provision for pumping final tailings slurry from the plant to the existing tailings facility via a new delivery pipe with spigots to facilitate an even distribution.

21.1.5 Infrastructure

The site already has an electrical power supply and access roads, so no provision is made for any off-site infrastructure. However, additional transformer capacity is required (as previous units were mostly removed). The new electrical equipment will all be pre-assembled off-site in a container(s) and then positioned on site. Also, a small amount of capital has been allocated for minor upgrades to on-site roads.

Plant infrastructure provisions in the capex estimate include compressed air, clean, process, gland service and tailings return water circuits.

There are some existing offices on site which can be utilized, but some additional space will be available within a new plant building which will also house stores and workshop areas.

21.1.6 Owners Costs

A provision of CAD1 Million is included for Owner's costs (see Table 21.3).

Table 21.3
Breakdown of Owners Costs Provision

Expense	Cost (CAD)
Training – Operation/Maintenance Labour	280,000
Site Construction Management	195,000
H/Office Support – Expenses	120,000
Permitting	80,000
Drilling of Stockpile	250,000
Recruitment	35,000
Miscellaneous Disbursements	40,000
Total	**1,000,000**

- The training provision allows for 2 months for all personnel prior to commissioning (the potential for some government funding for this activity is being investigated).
- Site construction management provides for one site manager, one assistant manager and a clerk for a total of 9 months.
- Home office support expenses provides for travel and food during site visits by head office personnel - accommodation for Avalon personnel will be at Avalon's house, but a provision for a part-time housekeeper is included.
- Permitting – permits for final construction and operations.
- It is planned to complete a drilling exercise on the low-grade stockpile to assist with production scheduling and plant feed grade control.
- There will be a number of expenses associated with the recruitment of operating personnel and miscellaneous activities both at site (e.g., construction power) and at head office.

The Owner's costs estimate does not include labour costs for head office personnel as these will be covered by Avalon's normal corporate operating costs.

21.1.7 Contingency

A contingency of 20% has been added to the capital cost estimate. This is considered acceptable on the basis that significant detail has already gone into the process design and equipment sizing. In addition, there is tremendous potential for equipment savings through the procurement of second-hand equipment particularly for the mill, screens and gravity concentrators.

21.2 OPERATING COSTS

Operating costs have been determined by Avalon and reviewed by Micon, and are expressed in Canadian Dollars based on:

- Total tonnes mined as determined by the mining schedule, and typical industry rates.
- Anticipated labour complements and appropriate labour rates and burdens.
- Energy estimates calculated from electrical equipment loads and current tariffs.
- Estimates for miscellaneous minor operating expenses.
- Current costs for site management and water treatment.
- Reagent dosages from testwork programs.

A summary of the LOM average annual costs is presented in Table 21.4, followed by a more detailed breakdown in Table 21.5.

Table 21.4
Summary of Operating Costs

Category	Ave. Annual Costs (CAD'000)	CAD/t Milled	CAD/t Tin	CAD/t Conc.
Stockpile Reclaim & Mining	3,588	4.40	5,076	2,792
Concentrator Processing	6,556	8.04	9,274	5,102
Concentrate Transport	289	0.36	409	225
Remediation & Site Management	848	1.04	1,200	660
General & Administration	340	0.42	480	264
Total Production Costs CAD	**11,583**	**14.25**	**16,439**	**9,044**
Total Production Cost USD	**8,910**	**10.96**	**12,646**	**6,957**

Table 21.5
Detailed Breakdown of Estimated Annual Operating Costs
(CAD '000's)

EXPENSE	LOM	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
Mining																				
Stockpile/Ore Reclaim Contractor	7,337	504	130	93	58	112	96	134	138	177	104	118	123	116	175	1,052	1,052	1,052	1,052	1,052
Exploration	500		250	250																
Mining- ore	43,326	–	3,300	3,437	3,569	3,366	3,426	3,284	3,270	3,123	3,396	3,346	3,327	3,351	3,132	–	–	–	–	–
Mining- Waste	15,216	–	6,592	2,857	886	19	5	58	405	274	188	3,523	382	25	2	–	–	–	–	–
Crushing Contractor	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Mining Total	**66,380**	**504**	**10,272**	**6,638**	**4,513**	**3,498**	**3,527**	**3,476**	**3,813**	**3,574**	**3,688**	**6,986**	**3,832**	**3,493**	**3,308**	**1,052**	**1,052**	**1,052**	**1,052**	**1,052**
Concentrator																				
Labour	32,719	1,309	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745	1,745
Energy- Crushing Plant	5,950	161	322	322	322	322	322	322	322	322	322	322	322	322	322	322	322	322	322	322
Energy- Process Plant (exc. Crushing)	43,051	1,102	2,205	2,205	2,205	2,205	2,582	2,582	2,582	2,582	2,582	2,582	2,205	2,205	2,205	2,205	2,205	2,205	2,205	2,205
Miscellaneous Consumables	1,850	50	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Grinding Media + Liners	26,677	842	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,472	1,472	1,472	1,472	1,472
Flotation Reagents	1,893	5	9	9	9	9	296	296	296	296	296	296	9	9	9	9	9	9	9	9
Environmental Monitoring	1,900	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Maintenance	7,248	188	376	376	376	376	457	457	457	457	457	457	376	376	376	376	376	376	376	188
Plant Expansion	–																			
Concentrator Total	**120,588**	**3,656**	**6,177**	**6,177**	**6,177**	**6,177**	**6,923**	**6,923**	**7,023**	**7,023**	**7,023**	**7,023**	**6,277**	**6,277**	**6,277**	**6,328**	**6,328**	**6,328**	**6,328**	**6,140**
Concentrate & Site Management																				
Site Management Costs (current)	9,690	510	510	510	510	510	510	510	510	510	510	510	510	510	510	510	510	510	510	510
Transportation of Conc	5,350	111	395	312	318	315	331	328	298	303	328	280	291	290	293	231	231	231	231	231
Rehabilitation-Tailings Placement	6,000						500	500	500	500	500	500							1,000	2,000
Total Site & Concentrate	**21,040**	**621**	**905**	**822**	**828**	**825**	**1,341**	**1,338**	**1,308**	**1,313**	**1,338**	**1,290**	**801**	**800**	**803**	**741**	**741**	**741**	**1,741**	**2,741**
Genaral & Administration																				
Labour	3,469	139	185	185	185	185	185	185	185	185	185	185	185	185	185	185	185	185	185	185
Miscellaneous	2,813	113	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150
Site visits	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Consultation fees	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Genaral & Administration Total	**6,281**	**251**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**	**335**
Total operating costs	**214,289**	**5,032**	**17,689**	**13,972**	**11,854**	**10,835**	**12,126**	**12,072**	**12,479**	**12,245**	**12,384**	**15,634**	**11,245**	**10,906**	**10,723**	**8,456**	**8,456**	**8,456**	**9,456**	**10,269**

21.2.1 Stockpile Reclaim and Mining

A cost of CAD1.25/t moved has been used for the reclamation of material by front-end loader from the stockpile and delivered to a crusher feed bin. It is intended that the crushing plant operates initially on a single 12-hour shift per day.

Costs for mining of plant feed and waste rock from the pits have been determined using a rate of CAD4.70/t mined for the Baby Zone and the Main Pit North based on a budget proposal received from a local mining contractor. This price includes drilling, blasting and transport of mill feed to the crusher and waste rock to the waste rock storage facility.

The cost for mining in the Main Zone North and West is assumed to be CAD0.5/t lower due to the waste rock being deposited in the mined-out Main Zone North area, hence incurring a lower transport cost.

21.2.2 Concentrator

21.2.2.1 Power

Power costs have been determined based on the installed mechanical equipment load. It is assumed all operating drives draw 80% of their installed power except for certain intermittent operating equipment items such as filter circuits, samplers, spillage pumps and stand-by equipment. An average power cost of CAD0.099/kWh has been used based on current rates from the local power supplier.

21.2.2.2 Labour

Labour requirements for the processing plant are estimated at only 26 personnel as indicated in Table 21.6 below for a total annual operating cost of CAD1.93M. Operations will also be supported by Head Office personnel on a part-time basis for activities such as procurement, accounting, human resources and technical support in areas such as metallurgy, environmental, geology and marketing.

The “day-shift” operator will oversee on-site environmental monitoring, basic training, on-site safety, product packing and dispatch, etc. under the supervision of the operations manager.

The operations manager will have a strong process/operations background, and the millwright will also be responsible for maintenance planning and general engineering supervision.

Table 21.6
Breakdown of Operating Labour & Costs

Department	Position Title	Number	Rate CAD/y Estimate	Total Annual Costs (CAD)
Administration	Operation Manager	1	$140,000	140,000
Administration	Clerk	1	$45,000	45,000
Operations	Plant Operators	4	$75,000	300,000
Operations	Shift/Plant Foreman	4	$85,000	340,000
Operations	Crusher Plant Assistants	4	$70,000	280,000
Operations	Day Shift Operator	2	$70,000	140,000
Technical	Lab Technician	1	$55,000	55,000
Engineering	Mill Wright	1	$90,000	90,000
Engineering	Electrician	1	$80,000	80,000
Engineering	Fitter	1	$80,000	80,000
Engineering	Aides- Shift	4	$60,000	240,000
Engineering	Aides- Day	2	$70,000	140,000
Total		**26**		**1,930,000**

21.2.2.3 Reagents and Consumables

Reagent consumption is almost completely related to the dosing of lime for water treatment which is included in the CAD510,000 per annum under “Site Management” costs in Table 21.5 above.

There is also the need for a very small volume of flotation reagents for removal of sulphides from the initial gravity concentrate. Consumption then increases significantly for Years 6-11 when sulphides are being removed from the tailings. Reagent costs are based on testwork dosage rates and budget supply costs. These indicate a flotation reagent cost of CAD0.581/t of material treated. It is assumed that a total of 3 Mt of cleaned tailings are required for capping the dam and so operating costs are based on the flotation of 510,000 t/y of gravity tailings for a 6-year period (Years 6-11).

Process water is to be sourced from the draining of the 2 pits and from recycling of water from the water treatment plant hence no cost for supply of process water is included.

Other consumables include grinding media (calculated assuming 1.25 kg/t), mill liners (one complete set per annum at CAD162,500/set) and crusher liners (CAD100,000/annum).

21.2.2.4 Maintenance

Annual maintenance supply costs have been estimated using 3% of the installed mechanical equipment costs per annum. This will be predominantly for pump spares and wear components in the gravity concentrators.

21.2.2.5 Environmental and Tailings

There is a CAD100,000 per annum allowance for environmental monitoring which provides for general water trench maintenance/repairs as well as on-going analyses of various environmental samples. In addition, there is a CAD500,000/annum provision during Years 6-11 for capping of the tailings facility (based on the placement of 500,000 t/y at CAD1.00/t).

Final closure costs are expected to be relatively small as remediation of the tailings facility will have been completed during Years 6-11 and the balance of the tailings and waste rock will have been deposited in the 2 mined out pits. The plant building will also be specifically designed for easy removal and potential sale or relocation. Hence a provision of CAD3 Million spread across Years 18 and 19 is considered sufficient for this activity.

21.2.2.6 Transportation of Concentrate

A cost of CAD225/t of dry concentrate produced is assumed (based on budget quotations) for concentrate transport to customers in Malaysia or Indonesia.

22.0 ECONOMIC ANALYSIS

Micon has prepared this PEA of the Project on the basis of a discounted cash flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to determine the potential viability of the proposed development of the East Kemptville Tin Production and Site Remediation Project. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

22.1 MACRO-ECONOMIC ASSUMPTIONS

22.1.1 Exchange Rate, Inflation and Discount Rate

The tin price assumed is based on a USD rate, but unless otherwise stated, financial results are expressed in CAD. Cost estimates and other inputs to the cash flow model for the Project have been prepared using constant, second quarter 2018 money terms, i.e., without provision for escalation or inflation.

An exchange rate of CAD1.30/USD is applied in the base case, approximately equal to current rates and to the trailing average over the past two years.

Micon has applied a real discount rate of 8% in its base case evaluation, approximating the weighted average cost of capital (WACC) for the Project.

22.1.2 Expected Metal Prices

The base case cash flow projection assumes a constant price of USD21,038/t tin metal as forecast by the World Bank for 2020 (see Section 19.0 for further forecast information).

22.1.3 Taxation Regime

Nova Scotia mining taxes, and Canadian federal and provincial income taxes payable on the Project have been provided for in the cash flow forecast. Mining tax is charged at the greater of 2% of gross revenue and 15% of net income from the mine.

Provincial and federal income tax rates are 16% and 15%, respectively. Depreciation allowances for income tax are generally limited to 25% on a declining balance basis, with only a small proportion of initial capital assumed to be eligible for accelerated allowance that may be claimed during the transition period ending in 2020.

22.1.4 Royalty

No royalty has been provided for in the cash flow model.

22.1.5 Selling Expenses

A provision for concentrate transport from East Kemptville to customer of CAD225/t is included within forecast cash operating costs.

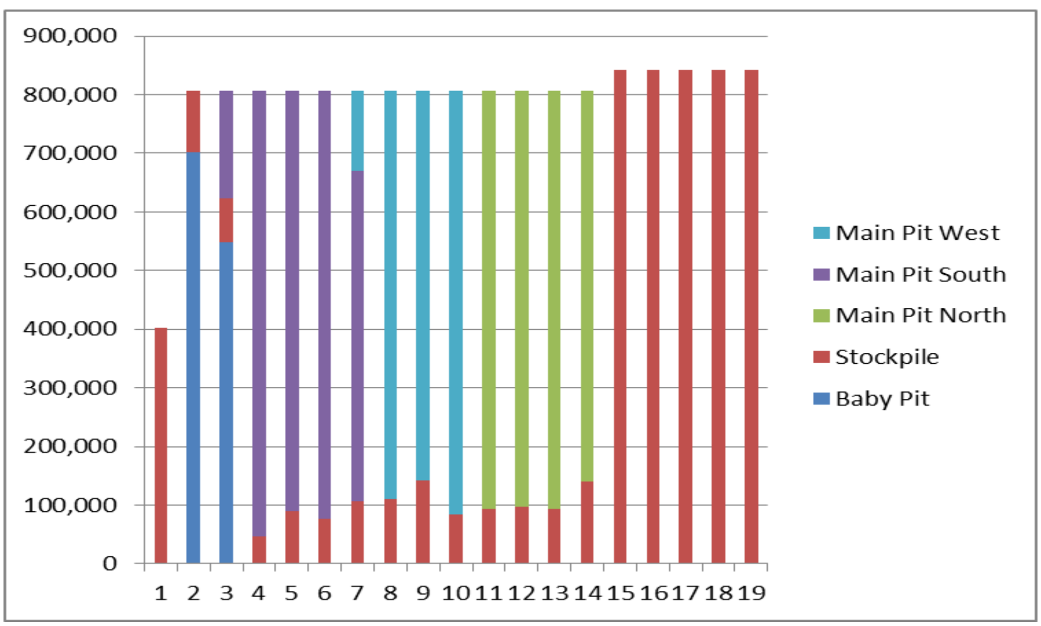
22.2 TECHNICAL ASSUMPTIONS

The technical parameters, production forecasts and estimates described elsewhere in this report are reflected in the base case cash flow model. These inputs to the model are summarized below. The measures used in the study are metric throughout.

22.2.1 Plant Feed and Tin Concentrate/Metal Production Schedule

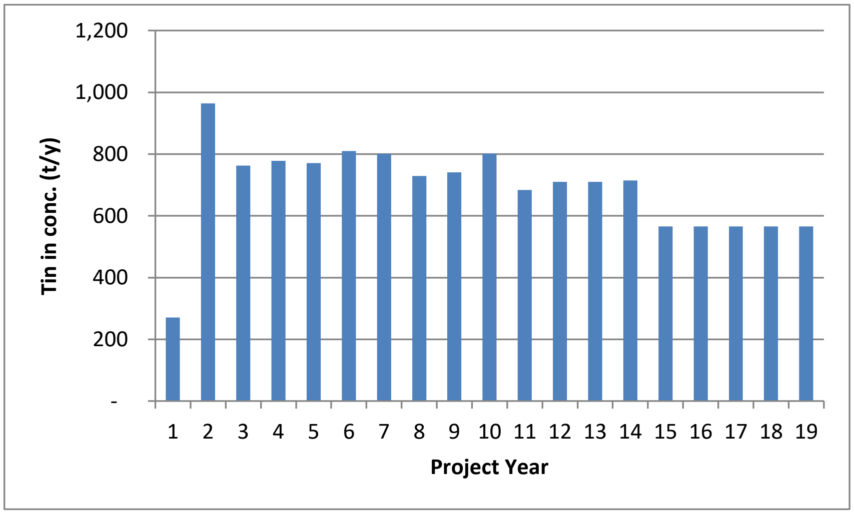
Figure 22.1 shows the annual tonnage of plant feed material reclaimed from the stockpile and mined from the two open pits.

Figure 22.1
Annual Plant Feed Production Schedule
(tonnes)



Annual production of tin metal (as a 55% tin concentrate) is shown in Figure 22.2.

Figure 22.2
Annual Tin Production Schedule



22.3 COSTS

22.3.1 Operating Costs

Cash costs over the Life-of-Mine (LOM) average CAD14.25/t milled (Table 21.4 and Table 21.5) which include all the costs associated with the maintenance and remediation of the tailings storage facilities.

22.3.2 Capital Costs

Pre-production capital expenditures are estimated to total CAD30.67 M which includes CAD19.96 M in Direct costs, CAD4.14 M Indirect costs, CAD0.75 M for buildings and tailings and CAD 1M for Owners Costs. A contingency of 20% on Directs and Indirects (CAD4.82M) is also included.

Not included in the Capital Cost estimate nor the Operating Cost figures is a provision of CAD850,000 during construction and Year 1 of operations for the dewatering of both the Baby Zone and Main Zone pits.

No sustaining capital is forecast, since all maintenance requirements are included in the Operating Cost estimate. This includes CAD5.0M for capping of the tailings facility plus CAD3M for final site remediation.

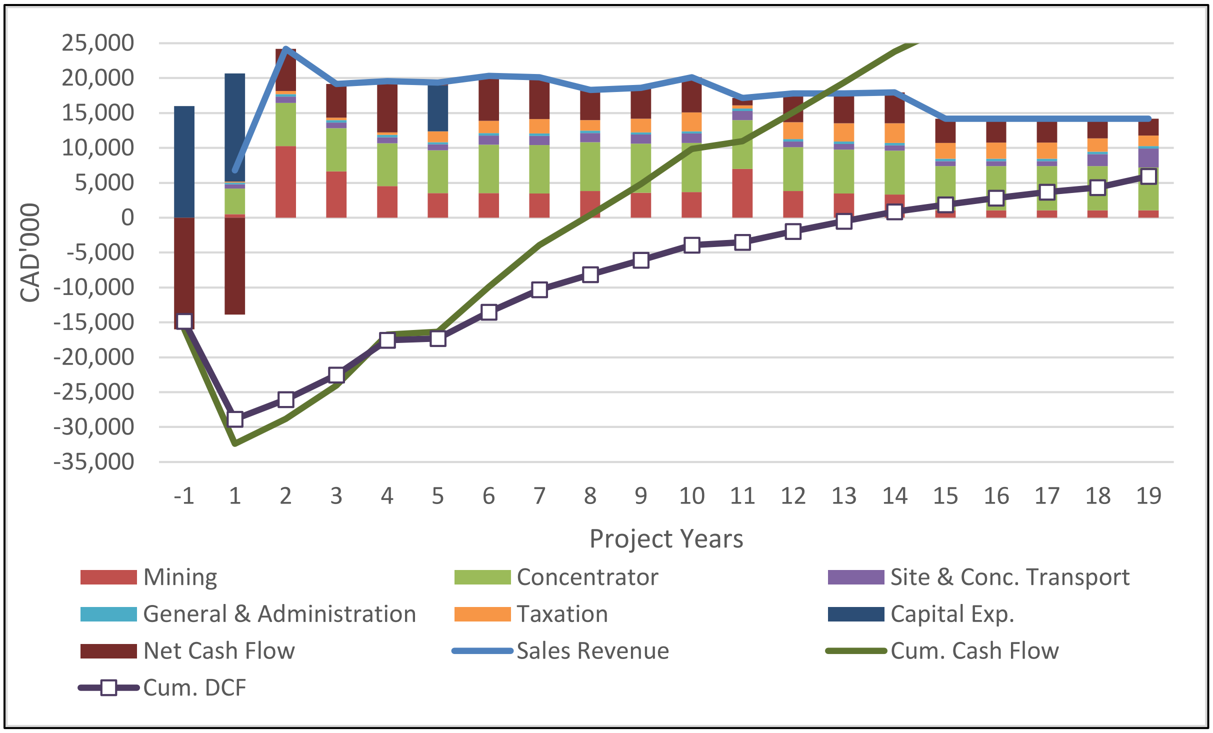
22.3.3 Base Case Cash Flow

The base case Project annual cash flows are presented in Table 22.1 and summarized in Figure 22.3.

Table 22.1
Life of Mine Annual Cash Flow

	Units	LOM Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Production																						
Mine:																						
Stockpiles Reclaimed	mt	5,870,000	-	403,000	103,820	74,670	46,675	89,854	77,075	107,348	110,311	141,589	83,496	94,121	98,120	92,958	139,632	841,466	841,466	841,466	841,466	841,466
Ore Mined		9,218,329		-	702,180	731,330	759,325	716,146	728,925	698,652	695,689	664,411	722,504	711,879	707,880	713,042	666,368	-	-	-	-	-
Strip Ratio		0.4x			2.0x	0.8x	0.2x	0.0x	0.0x	0.0x	0.1x	0.1x	0.1x	1.1x	0.1x	0.0x	0.0x					
Waste Mined		3,237,483		-	1,402,490	607,924	188,452	4,148	1,000	12,346	86,168	58,264	40,028	749,570	81,308	5,419	367	-	-	-	-	-
Total Mined		12,455,813		-	2,104,670	1,339,254	947,777	720,294	729,924	710,999	781,857	722,675	762,532	1,461,448	789,188	718,461	666,734	-	-	-	-	-
Ore to Plant	mt	15,088,329	-	403,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	841,466	841,466	841,466	841,466	841,466
Tailings	mt	15,064,552	-	402,508	804,246	804,613	804,584	804,599	804,528	804,543	804,674	804,653	804,542	804,756	804,709	804,710	804,700	840,438	840,438	840,438	840,438	840,438
Processing:																						
Crusher Feed	mt	15,088,329	-	403,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	841,466	841,466	841,466	841,466	841,466
Ball Mill Feed	mt	15,088,329		403,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	806,000	841,466	841,466	841,466	841,466	841,466
Ball Mill Feed Grade	% Sn	0.112%		0.112%	0.199%	0.158%	0.161%	0.159%	0.167%	0.166%	0.151%	0.153%	0.166%	0.141%	0.147%	0.147%	0.148%	0.112%	0.112%	0.112%	0.112%	0.112%
Tin Gravity Recovery	%	60.0%		60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%
Recovered Sn	mt	13,078		271	965	763	779	771	810	801	729	741	802	684	710	710	715	565	565	565	565	565
Tin Concentration Grade	% Sn	55.0%		55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%
Tin Concentrate Production	mt	23,777		492	1,754	1,387	1,416	1,401	1,472	1,457	1,326	1,347	1,458	1,244	1,291	1,290	1,300	1,028	1,028	1,028	1,028	1,028
Revenue from Sales																						
Tin Concentrate	CAD'000	328,303		6,799	24,216	19,154	19,545	19,347	20,328	20,117	18,313	18,601	20,127	17,179	17,830	17,818	17,953	14,196	14,196	14,196	14,196	14,196
TOTAL REVENUE FROM SALES	**CAD'000**	328,303		6,799	24,216	19,154	19,545	19,347	20,328	20,117	18,313	18,601	20,127	17,179	17,830	17,818	17,953	14,196	14,196	14,196	14,196	14,196
Costs																						
OPERATING COSTS																						
Mining	CAD'000	66,380		504	10,272	6,638	4,513	3,498	3,527	3,476	3,813	3,574	3,688	6,986	3,832	3,493	3,308	1,052	1,052	1,052	1,052	1,052
Concentrator	CAD'000	121,288		3,756	6,277	6,277	6,277	6,277	7,023	7,023	7,023	7,023	7,023	7,023	6,277	6,277	6,277	6,328	6,328	6,328	6,328	6,140
Site & Conc. Transport	CAD'000	21,040		621	905	822	828	825	1,341	1,338	1,308	1,313	1,338	1,290	801	800	803	741	741	741	1,741	2,741
General & Administration	CAD'000	6,281		251	335	335	335	335	335	335	335	335	335	335	335	335	335	335	335	335	335	335
TOTAL OPEX	**CAD'000**	214,989		5,132	17,789	14,072	11,954	10,935	12,226	12,172	12,479	12,245	12,384	15,634	11,245	10,906	10,723	8,456	8,456	8,456	9,456	10,269
EBITDA	CAD'000	113,314	0	1,667	6,428	5,081	7,591	8,411	8,101	7,945	5,834	6,356	7,743	1,544	6,586	6,913	7,230	5,739	5,739	5,739	4,739	3,927
TOTAL TAXES	CAD'000	31,163	0	136	484	383	391	1,362	1,716	1,999	1,471	1,940	2,697	460	2,416	2,634	2,825	2,253	2,285	2,308	1,912	1,490
NET INCOME AFTER TAXES	CAD'000	82,152	0	1,531	5,943	4,698	7,200	7,050	6,385	5,947	4,363	4,417	5,046	1,084	4,169	4,278	4,405	3,486	3,454	3,431	2,827	2,437
TOTAL CAPEX	CAD'000	38,138	16,006	15,512	-	-	-	6,620	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Working capital	CAD'000	-		(2,500)	(2,500)																-	5,000
Net Cash flow before tax	**CAD'000**	**75,176**	**-16,006**	**-16,346**	**3,928**	**5,081**	**7,591**	**1,791**	**8,101**	**7,945**	**5,834**	**6,356**	**7,743**	**1,544**	**6,586**	**6,913**	**7,230**	**5,739**	**5,739**	**5,739**	**4,739**	**8,927**
Cum. Cash Flow		**IRR: 15.0%**	-16,006	-32,351	-28,424	-23,342	-15,751	-13,960	-5,859	2,087	7,920	14,277	22,019	23,564	30,149	37,062	44,292	50,031	55,771	61,510	66,249	75,176
Payback (undisc.)	yrs	6.7		1.0	1.0	1.0	1.0	1.0	1.0	0.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Discounted cash flow (DCF) before tax	**8%**	**17,861**	-14,820	-14,014	3,118	3,735	5,166	1,129	4,727	4,293	2,918	2,944	3,321	613	2,422	2,353	2,279	1,675	1,551	1,436	1,098	1,915
Cum. DCF	CAD'000		-14,820	-28,834	-25,716	-21,981	-16,815	-15,686	-10,959	-6,666	-3,748	-804	2,517	3,130	5,552	7,905	10,185	11,860	13,411	14,847	15,945	17,861
Payback (disc.)	yrs	9.2		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Net Cash flow after tax	**CAD'000**	**44,013**	**-16,006**	**-16,482**	**3,443**	**4,698**	**7,200**	**430**	**6,385**	**5,947**	**4,363**	**4,417**	**5,046**	**1,084**	**4,169**	**4,278**	**4,405**	**3,486**	**3,454**	**3,431**	**2,827**	**7,437**
Cum. Cash Flow		**IRR: 10.6%**	-16,006	-32,487	-29,044	-24,346	-17,146	-16,716	-10,330	-4,384	-21	4,396	9,442	10,525	14,695	18,973	23,378	26,864	30,318	33,749	36,577	44,013
Payback (undisc.)	yrs	8.0		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Discounted cash flow (DCF) after tax	**8%**	**5,608**	-14,820	-14,130	2,733	3,453	4,900	271	3,726	3,213	2,182	2,046	2,164	430	1,533	1,457	1,389	1,018	934	859	655	1,596
Cum. DCF	CAD'000		-14,820	-28,950	-26,217	-22,764	-17,863	-17,593	-13,867	-10,654	-8,472	-6,426	-4,262	-3,831	-2,298	-842	547	1,565	2,498	3,357	4,012	5,608
Payback (disc.)	yrs	13.6		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.6	0.0	0.0	0.0	0.0	0.0

Figure 22.3
Annual Cash Flow



This PEA is preliminary in nature; it 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.

Before tax, the base case demonstrates an undiscounted payback period of 6.7 years, and an IRR of 15.0%. At an annual discount rate of 8%, the Project has a net present value (NPV_8) before tax of CAD17.8 million, and the payback period extends to 9.2 years.

After tax, the base case undiscounted payback period is 8.0 years, leaving a tail of 11 years planned production, and the Project has an IRR of 10.6%. At an annual discount rate of 8%, the Project NPV_8 after tax is CAD5.6 million, and the payback period extends to 13.6 years.

22.4 SENSITIVITY STUDY

22.4.1 Capital, Operating Costs, Tin Price and Recovery Sensitivity

The sensitivity of Project returns to changes in capital, operating costs and all revenue factors (including recovery and tin price) was tested over a range of 20% above and below base case values (Figure 22.4 and Figure 22.5).

Figure 22.4
NPV Sensitivity Diagram

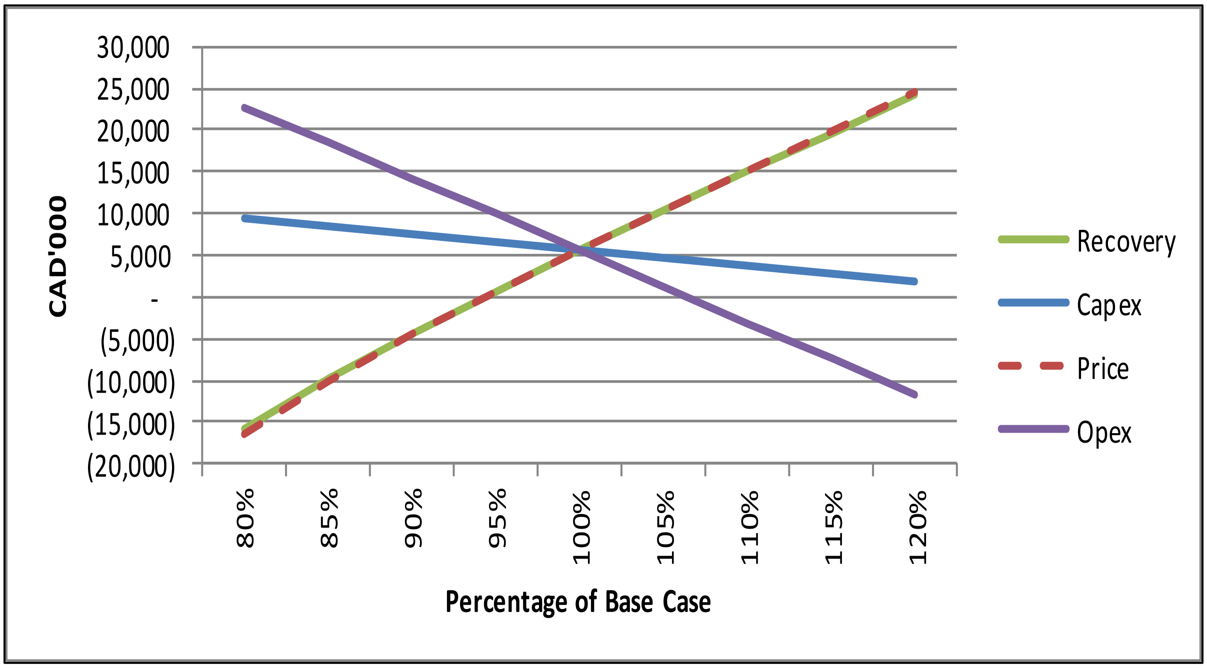
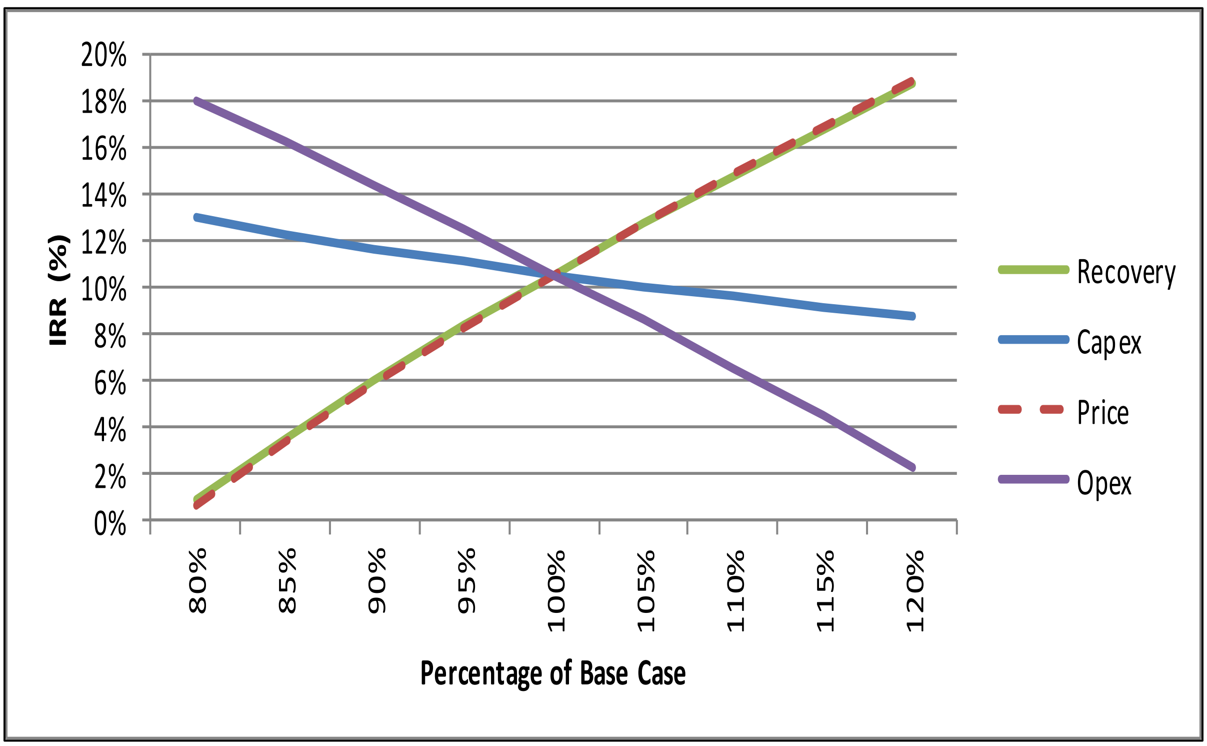


Figure 22.5
IRR Sensitivity Diagram



The charts suggest that the Project is most sensitive to revenue drivers, namely tin price and recovery which are essentially identical, with an adverse change of 5% in tin price resulting in a reduction of almost CAD5 million in NPV_8. The Project is somewhat less sensitive to changes in operating cost, with a 7.5% increase in costs resulting in a near-zero NPV_8. The Project is least sensitive to capital cost, with NPV_8 remaining positive even with a 20% increase in capital costs.

22.5 CONCLUSION

Avalon has the opportunity to re-commence commercial tin production from the East Kemptville mine by establishing a small-scale operation processing an on-surface, Low Grade Stockpile and higher grade, near surface occurrences within the existing pits. Assuming a tin price of USD21,038/t (as forecast by the World Bank Commodity Price for 2020), and an exchange rate of CAD1.30/USD, the Project has an indicated pre-tax IRR of 15.0% and an NPV_8 of CAD17.9 million. After tax IRR is 10.6% and NPV_8 after tax is CAD5.6 million. Initial capital cost is estimated at CAD31.5 million which the after-tax cash flow pays back in approximately 8 years, leaving a tail of 11 years planned production.

In so -doing, Avalon also has the opportunity to remediate the site by removing the existing environmental liabilities created by previous operations.

While the results of the PEA indicate economic potential, there are a number of opportunities to further improve Project economics. One of the most promising of these is the potential to upgrade the feed material to the processing plant through ore-sorting. Results from an initial evaluation of ore-sorting technology carried out in 2017 were very encouraging. Additional results from a second evaluation using an alternative ore-sorting technology are expected by the end of this month, after which further testwork or a piloting program at site is likely.

Successful application of ore-sorting process technology offers a number of potential benefits to the Project financial model. By rejecting non-mineralized waste rock before feeding it into the concentrator, ore-sorting allows for a reduction in the size of the concentrator with attendant reductions in both capital and operating costs. It may also allow for economic recovery of tin from other mineralized materials stored on site that are presently not included in the re-development model.

23.0 ADJACENT PROPERTIES

Avalon holds exploration claims adjacent to the East Kemptville mine site as disclosed in Section 4.0 of this report. Relevant information on these claims is to be found in Sections 4.0 and 6.0 of this report.

There are no third-party exploration licences adjacent to Avalon's Special Licence.

24.0 OTHER RELEVANT DATA AND INFORMATION

Micon and Avalon believe that no additional information or explanation is necessary to make this Technical Report understandable and not misleading. Any requests for clarification should be addressed to Avalon at office@AvalonAM.com.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 INTERPRETATION

The study has determined that processing of the stockpiled material and higher-grade zones from the pits can be economically viable across an operating life of 18.5 years producing a total of approximately 24,000 tonnes of 55% tin concentrate (13,078 t of contained tin).

The initial capital cost estimate for the Project is CAD31.5 M with a further CAD6.6 M in year 5 of operations for a flotation plant to remove sulphides and produce clean tailings for capping of the TMF. There is no sustaining capital indicated as all such costs (additional plant, tailings maintenance, etc.) are included in the operating cost estimates.

The simple processing circuit proposed has minimal reagent requirements, a low power demand (~29-32,000 MWh/y) and small operating labour force (26 full time personnel in total) resulting in a low cost of production. It also has the potential to commence production relatively quickly (approximately 18-20 months) as the lead time on the processing plant is short due to its' scale and simplicity, plus permitting requirements are not overly onerous.

The Project economics are sufficiently robust to accommodate some fluctuation in pricing and the tin metal price and there remains the potential to also recover concentrates of copper and zinc/indium should the prices of these metals warrant doing so.

The capital cost estimate is based on 100% new equipment so there are a number of opportunities to reduce this cost by purchasing good quality used equipment.

A well-thought-out remediation program has been developed which has been scrutinized by a number of independent parties as well as local government authorities. This program will enable Avalon to fully demonstrate its commitment to sustainable "mining" through responsible actions as it restores the site to one with no on-going environmental liabilities.

25.2 RISKS AND OPPORTUNITIES

The Project in its current format has the following risks and opportunities.

25.2.1.1 Head Grade to Mill

Most of the area of the Main Zone deposit is drilled at 25 m intervals along lines 100 m apart. The mineralization at East Kemptville has a wide range of grades that can change in short distances. Due to the "nugget effect" that this implies, there is risk of difficulty of accurately predicting mill feed grade during mining. As the objective is mining and processing modest tonnages of higher grade sections of the deposit, which suggests selective mining, it is prudent to consider infill diamond drilling specific areas of the proposed pits to increase the resource confidence, especially of Inferred Resources but also Indicated resource category sections. The opportunity presented by the drill hole spacing is that there may be areas of potential high-grade mining that are poorly defined and unrecognised at present due to the wide drill hole spacing thus increasing the mine life and financial return. The operating cost schedule provides CAD250,000 in each of Years 2 and 3 for conducting suitable drill programs within both pits once they are dewatered.

25.2.1.2 Resources

Opportunities exist to increase resources for the Project. For example, it is believed that a more detailed mine plan is likely to increase the tonnage of high grade, near surface material in these pits which could extend the life of the operation.

There has been no examination of the possibility of underground mining. Deep drilling on the Baby Zone has suggested that tin mineralization continues close to 100 m below the bottom of the presently planned pits. A detailed examination of this data may reveal underground mining potential in this and other areas of the property.

In Section 9.1, opportunities for additional resources are discussed, including the Duck Pond Zone and area west of the Baby Pit.

Further, there are additional very low-grade stockpiles on surface which could potentially be processed if methods such as ore-sorting are demonstrated to have the ability to pre-concentrate the tin prior to the milling circuit

25.2.1.3 Tin Price

The Project is somewhat sensitive to the tin price but can accommodate a ~33% drop in price (i.e. from USD21,038/t to ~USD14,100/t) before cost/revenue breakeven (Revenue from Sales equals Operating Costs). An analysis of recent historical tin prices indicates that the LME listed price for tin has been above that value virtually continuously for more than the past 10 years. The LME listed price as of 1st May 2018 is USD21,395 and the World Bank Commodity Price forecast indicates tin a long-term price forecast of USD20,169 for 2025.

25.2.1.4 Tin Recovery

Economic sensitivity to tin recovery is identical to that for tin price. The recovery of 60% is based on the testwork program by Met-Solve, and Avalon believes that once the plant is up and running, this figure can actually be improved upon. The reason for this is that with the bench scale testwork, it is very difficult to simulate the impact of recirculating streams and optimize recovery over time so material that would be captured from such streams often reports to tailings during bench testing. With an operating plant, these streams are fully recycled, and operators have the opportunity to optimize recovery. The Project can accommodate a drop-in recovery to around 40% before cost/revenue breakeven.

25.2.1.5 Mining

The forecast mining costs represent almost 30% of total production costs and are estimated using typical industry contractor rates for open pit operations of this size. Upon completion of the proposed drilling to update the resource model, further mine design work and haulage analyses are required before costing of the final tonnages of material (plant feed plus waste) to be mined can be more accurately defined.

25.2.1.6 Stockpile Grade

The grade of the material in the stockpile has been estimated by two surface sampling programs and by reviewing historical information, all of which produced similar results, and as a consequence an "inferred" resource has been determined by an external consultant. It is, however, planned to complete a drill program of the stockpile as soon as financing is available, partly to confirm the overall grade, but more importantly to map the internal grade distributions and produce a more representative schedule of feed grades shipped to the processing plant from this source.

25.2.1.7 Operating Life

The current operating life is 18.5 years; however, Avalon is confident that additional feed sources will be identified, and that the operating life will be extended by:

- Processing other, lower grade deposits on site (installation of a dense media separation or ore-sorting circuit as a pre-concentration step could make these viable).
- The "Duck Pond" area is known to contain tin mineralization that could be exploited although further drilling is required to determine plant feed potential here. Historic resources for Duck Pond have been reported at 9 Mt grading 0.11% Sn (Kooiman, 1989). In addition, drilling by Avalon at Duck Pond includes intercepts such as 0.86% Sn over 7.5 m (DPAV-15-22) and 0.60% Sn over 10.5 m (DPAV-15-24), implying potential for higher grade resources.
- "South Grid Area" has intercepts of tin mineralization reported in historic drill hole 90-010 with 0.17% Sn over 34.3 m and 90-008 with 0.31% Sn over 33.0 m in the granite. These intercepts are 700 m southwest of the Baby Pit. These intercepts have not been investigated further.
- Identifying additional higher-grade zones in the pits just below those areas planned to be mined in the PEA.
- Toll-treatment of other material from external operations.
- Moving to an underground mining operating if economics can be supported by a strong tin price.

25.2.1.8 Purchasing Used/Refurbished Equipment

The capital cost estimate has assumed all equipment is purchased new, but there are significant opportunities to reduce equipment costs, particularly for the crushers and mill, by purchasing used/refurbished items. Avalon is also aware of a number of used screens and gravity concentrators that could potentially be acquired.

25.2.1.9 Revenue from By-products

No provision has been made for up-grading the sulphide concentrate into marketable copper and zinc/indium concentrates for sale. However, should recent increases in the price of all 3 of these metals continue then this situation could change. It is possible that this mixed product may have some economic value if sold to one of the eastern Canadian base metal mining operators.

25.2.1.10 Foreign Exchange Rate

A lot of the mechanical equipment is being sourced from outside Canada and is priced in American dollars. Similarly, all revenue is in USD. An exchange rate of CAD1.30:USD1 has been used. Should the Canadian dollar strengthen this would be positive in terms of initial capex, but then negative with respect to subsequent revenue once in production.

25.2.1.11 Environmental Liability

By re-activating the Project, Avalon will be inheriting a number of (currently) long term environmental liabilities. However, by removing the low-grade stockpile, capping the tailings facility and depositing the balance of the tailings along with waste rock into the two pits, Avalon believes a "walk-away" closure strategy has been developed, eliminating these long-term liabilities.

Under a Memorandum of Understanding, Avalon and RAL/BHP have agreed in principle to work toward a mutually beneficial transition arrangement to minimize Avalon's up-front exposure to the existing environmental liability, while providing RAL/BHP with a pathway toward eliminating its exposure to the long-term liability. Such an agreement could include allowing Avalon access to already filed financial assurance in the short term and phasing this out as the Project progresses and once Avalon income is being generated. There will also be a reduction in the required financial assurance associated with the progressive remediation once the clean tailings cover on the existing CTP is established and the low-grade stockpile is no longer a source of AMD.

25.3 CONCLUSIONS

Avalon has the opportunity to re-commence commercial tin production from the East Kemptville mine by establishing a small-scale operation processing an on-surface, low-grade stockpile and higher grade, near surface occurrences within the existing pits. Assuming a tin price of USD21,038/t (as forecast by the World Bank Commodity Price indicates for 2020), and an exchange rate of CAD1.30/USD, the Project has an indicated pre-tax IRR of 15.0% and an NPV_8 of CAD17.8 million. After-tax IRR is 10.6% and NPV_8 after tax is CAD5.6 million. Initial capital cost is estimated at CAD31.5 million which the after-tax cash flow pays back in approximately 8 years, leaving a tail of 11 years planned production.

Avalon considers the tin concentrate produced (Table 25.1) to be highly marketable. In early 2018, Avalon entered into a non-binding MOU for the sale of all its production with a well-known company that owns a large tin smelter. The formula used by this customer for determining concentrate pricing has been used by Avalon in the financial model.

Table 25.1
Final Tin Concentrate Analysis

Element	**Sn**	**Cu**	**Zn**	**Fe**	**S**	**Pb**	**As**	**Cd**
Value (%)	55.22	0.009	0.014	0.57	0.08	0.005	0.002	<0.0001
Element	**Ni**	**Co**	**Bi**	**Hg**	**Se**	**SiO_2**	**Mn**	**CaF_2**
Value (%)	0.006	<0.001	<0.0001	<0.0001	0.0001	9.04	0.35	0.55

The re-development model, as presently conceived, is an environmental remediation Project that will be financed through the sale of tin concentrates recovered in large part from previously-mined mineralized material on the site.

From Day 1 of operations, Avalon’s model provides for a reduction in the long-term environmental liability and eventual full rehabilitation of this brownfield site. By processing the stockpiles, Avalon will be also removing a significant on-going source of acid rock drainage and contribute to the environmental remediation of the site. The intention will be to create a clean, re-habilitated “walk-away” site once operations are ceased.

The Project enjoys strong support from the community as well as from local politicians, First Nations and environmental NGOs. Avalon is also in discussions with a number of local businesses towards collaboration on future opportunities including, among others, a long-term vision for re-development of the rehabilitated site.

The start of operations is not anticipated to be subject to approvals under the Canadian Environmental Assessment Act 2012 (CEAA) as the mine does not exceed any of the CEAA triggers including mine and mill tonnages. The Project will not have any new impacts to fish or fish habitat, nor will it impact on any Federal Wildlife Areas or Migratory Bird Sanctuaries. Final Permitting and Approval for the Project is therefore expected to be relatively short and simple.

26.0 RECOMMENDATIONS

26.1 OVERVIEW

The preliminary economic assessment presents an attractive Project and the opportunity to generate significant revenue for Avalon as well as remediating an environmental problem. It is recommended therefore that the Project continues to the next stage of development.

26.2 RECOMMENDATIONS FOR THE NEXT PHASE OF PROJECT DEVELOPMENT

26.2.1 Resources

- The low-grade stockpile should be drilled, sampled and assayed to increase the confidence of the mineral resource estimate from an inferred category. This work will also confirm the overall grade and metal distribution so that a detailed plant feed schedule can be determined. Not only will this assist with metallurgical performance, it may also identify areas of abnormally high or low-grade zones which could require differing processing requirements (e.g., extremely low-grade zones could be left untreated).
- Once de-watered, a program of infill drilling is recommended for the Main and Baby zones in order to improve the geological data base and to improve understanding of the controls on mineralization and variability of grade. Also, it is likely there are other areas of shallow, high grade material which could be added to the feed stock particularly if the tin price continues to trend upwards. There are currently no plans to recover copper or zinc/indium concentrates, but if the recent increases in the prices of these metals continue then at some point it may become viable to extract them, especially when processing the high-grade material from the pits and even the coarse rocks already mined. Regular economic viability calculations should be performed/up-dated particularly in the early years prior to commencement of mining from the pits. Some additional drilling of the in-situ deposits might also be considered if metal prices improve and remain stable.
- During the course of operations, additional exploration should be conducted on other areas within and adjacent to the current property boundary in order to identify additional resources (e.g., Duck Pond area where prospective economic mineralization has already been identified).

26.2.2 Mining

- The mine designs and project schedules should be completed to a more detailed level using the revised mineral resources resulting from the work recommended above.
- Mining contractors should be requested to provide a more detailed mining contract proposal using these detailed mine plans and schedules.
- The economic potential of mining deeper (either through open pit or underground methods) should be investigated for the Main and Baby Zone mineralization

26.2.3 Processing Plant

- During the next phase of engineering, the proposed modular off-site fabrication and assembly philosophy should be adhered to as it will not only keep the up-front capital cost lower than normal but will also facilitate either future expansion or plant relocation to elsewhere once the East Kemptville resources have been exhausted.
- There is an opportunity to run a short pilot campaign to assess and optimize the initial "rougher" tin recovery performance. The purpose of this work will be to further optimize the grinding and classification circuit in order to minimize over-grinding of cassiterite. . The pilot rougher flotation circuit operation will also provide an opportunity to optimize performance and confirm tin recovery.
- The potential for using ore-sorting to upgrade the plant feed should be further investigated. This could have significant impacts on capital and/or operating costs either through the use of a smaller, cheaper processing plant or by significantly increasing the tin output through the same plant but over a shorter time frame. The pre-treatment by ore-sorting, of the "very low" grade stockpiles may also generate a suitably graded material to allow plant operations over a longer period.

26.2.4 Project Implementation

- The current 16-18-month implementation schedule is tight, and where possible, development activities should continue whilst project funding is being secured. Such activities could include finalizing fixed equipment prices, confirming fabricators to be used and negotiating various service and supply contracts.
- There are various minor permitting studies which still need to be completed in order to gain site access for initiating construction activities. These studies are a priority in order to prevent any possible impact on the implementation schedule.
- Securing a final agreement with BHP still needs to be completed but this must be subject to finalizing a mutually beneficial transition arrangement to minimize Avalon's up-front exposure to the existing environmental liability.
- The start of operations is not anticipated to be subject to approvals under the Canadian Environmental Assessment Act 2012 (CEAA) as the mine does not exceed any of the CEAA triggers including mine and mill tonnages. The Project will not have any new impacts to fish or fish habitat, nor will it impact on any Federal Wildlife Areas or Migratory Bird Sanctuaries. Final Permitting and Approval for the Project is therefore expected to be relatively short and simple.

26.3 BUDGET

The budget prepared by Avalon for the next phase of work to develop the East Kemptville Project towards production is presented below.

Table 26.1
Budget for the Next Phase of Project Development

Proposed Work	Cost (CAD)
Drilling and Resources Update	
Drilling Stockpile and Updated Resource Estimate	250,000
Economic Study Update	
Mini Pilot Plant Trial	100,000
Preliminary Engineering and detailed cost estimates	300,000
Updated economic study and NI 43-101 report	100,000
Environmental	
General studies and permitting applications	100,000
Total Proposed Budget (all items)	**850,000**

Micon has reviewed Avalon's budget for the next phase of work on the East Kemptville Project and considers it to be reasonable.

27.0 DATE AND SIGNATURE PAGE

"Richard Gowans" {signed and sealed as of report date}

Richard Gowans, P.Eng.	Report Date: August 30, 2018
President & Principal Metallurgist	Effective Date: July 24, 2018

"Christopher Jacobs" {signed and sealed as of report date}

Christopher Jacobs, CEng., MIMMM	Report Date: August 30, 2018
Vice President & Mining Economist	Effective Date: July 24, 2018

"Dayan Anderson" {signed and sealed as of report date}

Dayan Anderson, MMSA	Report Date: August 30, 2018
Senior Mining Engineer	Effective Date: July 24, 2018

"Jane Spooner" {signed and sealed as of report date}

Jane Spooner, P.Geo.	Report Date: August 30, 2018
Vice President	Effective Date: July 24, 2018

"William Mercer" {signed and sealed as of report date}

William Mercer, P. Geo.	Report Date: August 30, 2018
	Effective Date: July 24, 2018

"Donald Hains" {signed and sealed as of report date}

Donald Hains, P.Geo.	Report Date: August 30, 2018
	Effective Date: July 24, 2018

"Reid Smith" {signed and sealed as of report date}

Reid Smith, P.Geo.	Report Date: August 30, 2018
	Effective Date: July 24, 2018

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World Bank Group Commodity Markets Outlook (April 2018

World Bank Commodity Price Data (The Pink Sheet), and Commodity Markets Outlook April 2018.

29.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON
Richard Gowans, P.Eng.

As the co-author of this report for Avalon Advanced Materials Inc. (Avalon) entitled "The East Kemptville Tin Production and Site Remediation Project Preliminary Economic Assessment, Nova Scotia, Canada", effective date July 24, 2018, I, Richard Gowans, do hereby certify that:

1. I am employed as the President and Principal Metallurgist by, and carried out this assignment for Micon International Limited, Suite 900, 390 Bay Street Toronto, Ontario, M5H 2Y2. tel. (416) 362-5135 fax (416) 362-5763 e-mail: rgowans@micon-international.com
2. I hold the following academic qualifications:

   B.Sc. (Hons) Minerals Engineering, The University of Birmingham, U.K., 1980
3. I am a registered Professional Engineer of Ontario (membership number 90529389); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum.
4. I have worked as an extractive metallurgist in the minerals industry for over 35 years. Throughout my career I have worked on and managed a wide assortment of feasibility studies and technical audits on international industrial mineral, precious and base metal projects.
5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the management of technical studies and design of numerous metallurgical testwork programs and metallurgical processing plants.
6. I have not visited the Property that is the subject of this report.
7. I am responsible for Sections 1, 2, 3, 4, 5, 13, 17, 18, 21, 25 and 26 of this Technical Report.
8. I am independent of Avalon and related entities, as defined in Section 1.5 of NI 43-101.
9. I have not worked on or been associated with the Property prior to this Technical Report.
10. I have read NI 43-101 and the Sections of this report for which I am responsible have been prepared in compliance with the instrument.
11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Report Date: August 30, 2018
Effective Date: July 24, 2018

"Richard Gowans" {signed and sealed as of report date}

Richard Gowans, B.Sc., P.Eng.
President & Principal Metallurgist

CERTIFICATE OF QUALIFIED PERSON
Christopher Jacobs, CEng, MIMMM

As the co-author of this report for Avalon Advanced Materials Inc. (Avalon) entitled "The East Kemptville Tin Production and Site Remediation Project Preliminary Economic Assessment, Nova Scotia, Canada", effective date July 24, 2018, I, Christopher Jacobs, do hereby certify that:

1. I am employed as a Vice President and Mining Economist by, and carried out this assignment for, Micon International Limited, 900 – 390 Bay Street, Toronto, Ontario M5H 2Y2. tel. (416) 362-5135, email: cjacobs@micon-international.com.
2. I hold the following academic qualifications:

   B.Sc. (Hons) Geochemistry, University of Reading, 1980;

   M.B.A., Gordon Institute of Business Science, University of Pretoria, 2004.
3. I am a Chartered Engineer registered with the Engineering Council of the U.K.

   (registration number 369178).
4. Also, I am a professional member in good standing of: The Institute of Materials, Minerals and Mining; and The Canadian Institute of Mining, Metallurgy and Petroleum (Member).
5. I have worked in the minerals industry for more than 35 years; my work experience includes 10 years as an exploration and mining geologist on gold, platinum, copper/nickel and chromite deposits; 10 years as a technical/operations manager in both open-pit and underground mines; 3 years as strategic (mine) planning manager and the remainder as an independent consultant when I have worked on a variety of deposits including cobalt, copper and gold.
6. I have not visited the Property that is the subject of this report.
7. I am responsible for Section 22 of this Technical Report.
8. I am independent of Avalon and related entities, as defined in Section 1.5 of NI 43-101.
9. I have not worked on or been associated with the Property prior to this Technical Report.
10. I have read NI 43-101 and the Sections of this report for which I am responsible have been prepared in compliance with the instrument.
11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Report Date: August 30, 2018
Effective Date: July 24, 2018

"Christopher Jacobs" {signed and sealed as of report date}

Christopher Jacobs, CEng, MIMMM
Vice President & Mining Economist

CERTIFICATE OF QUALIFIED PERSON
Dayan Anderson, M.S., MMSA(QP)

As the co-author of this report for Avalon Advanced Materials Inc. (Avalon) entitled "The East Kemptville Tin Production and Site Remediation Project Preliminary Economic Assessment, Nova Scotia, Canada", effective date July 24, 2018, I, Dayan Anderson, do hereby certify that:

1. I am the Principal of Onyx Mining Services, California, and I carried out this assignment as an associate for Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario, M5H 2Y2 e-mail: danderson@micon-international.com.
2. I hold the following academic qualifications:

   B.S. Mining Engineering, Colorado School of Mines (1997)

   M.S. Environmental Studies, Green Mountain Collage (2018)
3. I am a Qualified Professional (QP) Member of the Mining & Metallurgical Society of America (MMSA). I am also a member in good standing of the Society for Mining, Metallurgy, and Exploration.
4. I have worked in the minerals industry for over 20 years across the coal, iron ore, industrial minerals, and base and precious metals sectors.
5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 2 years in ore control and operations, 6 years in strategic and tactical mine planning, 3 years of permitting and environmental management, 1 year as mine manager, and the remainder as an independent consultant working on technical studies, operational audits and due diligence assessments.
6. I have not visited the Property that is the subject of this report.
7. I am responsible for Sections 15,16, and portions of Sections 20.3 and 20.4 pertaining to the integrated mine, waste backfill and tailings deposition strategy of this Technical Report.
8. I am independent of Avalon and related entities, as defined in Section 1.5 of NI 43-101.
9. I have not worked on or been associated with the Property prior to this Technical Report.
10. I have read NI 43-101 and the Sections of this report for which I am responsible have been prepared in compliance with the instrument.
11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Report Date: August 30, 2018
Effective Date: July 24, 2018

"Dayan Anderson" {signed and sealed as of report date}

Dayan Anderson, M.S., MMSA(QP)
Senior Mining Engineer

MICON INTERNATIONAL LIMITED | mineral industry consultants

CERTIFICATE OF QUALIFIED PERSON
Jane Spooner, M.Sc., P.Geo.

As the co-author of this report for Avalon Advanced Materials Inc. (Avalon) entitled "The East Kemptville Tin Production and Site Remediation Project Preliminary Economic Assessment, Nova Scotia, Canada", effective date July 24, 2018, I, Jane Spooner, do hereby certify that:

1. I am employed as an Associate Specialist in Mineral Market Analysis and carried out this assignment for

   Micon International Limited
   Suite 900, 390 Bay Street
   Toronto, Ontario M5H 2Y2
   tel. (416) 362-5135 fax (416) 362-5763
   e-mail: jspooner@micon-international.com

2. I hold the following academic qualifications:

   B.Sc. (Hons) Geology, University of Manchester, U.K. 1972

   M.Sc. Environmental Resources, University of Salford, U.K. 1973

3. I am a member of the Association of Professional Geoscientists of Ontario (membership number 0990); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I have worked as a specialist in mineral market analysis for over 30 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the analysis of markets for base and precious metals, industrial and specialty minerals, coal and uranium; project due diligence assessments and project management.

6. I have not visited the Property that is the subject of this report.
7. I am responsible for Section 19 of this Technical Report.
8. I am independent of Avalon and related entities, as defined in Section 1.5 of NI 43-101.
9. I have not worked on or been associated with the Property prior to this Technical Report.
10. I have read NI 43-101 and the Sections of this report for which I am responsible have been prepared in compliance with the instrument.
11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Report Date: XXX XX, 2018
Effective Date: July 24, 2018

"Jane Spooner" {signed and sealed as of report date}

Jane Spooner, M.Sc., P.Geo.
Vice President

CERTIFICATE OF QUALIFIED PERSON
William Mercer, P. Geo.

As the co-author of this report for Avalon Advanced Materials Inc. (Avalon) entitled "The East Kemptville Tin Production and Site Remediation Project Preliminary Economic Assessment, Nova Scotia, Canada", effective date July 24, 2018, I, William Mercer, do hereby certify that:

1. I am employed as the Vice President, Exploration by Avalon Advanced Materials Inc. (Avalon) at Suite 1901, 130 Adelaide Street West, Toronto, Ontario, M5H 3P5, telephone 416 364 4938, email bmercer@avalonam.com

2. I hold the following academic qualifications:

   BSc (Geology), Edinburgh University, 1968

   PhD (Geology), McMaster University, 1975

3. I am a member of the Association of Professional Geoscientists of Nova Scotia (membership number 166) and the Association of Professional Geoscientists of Ontario (membership number 0186).

4. I have worked in mineral exploration for over 40 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes field exploration and senior level supervision as manager and chief geologist of exploration in Canada, Latin America and elsewhere for commodities including base and precious metals, industrial and specialty minerals, bauxite and uranium; operating mine due diligence assessments and project management.
6. I visited the Property that is the subject of this report on numerous occasions between 2014 and 2016.
7. I am responsible for Sections 6 to 12, 14, 23 and 24 of this Technical Report.
8. I am not independent of Avalon and related entities, as defined in Section 1.5 of NI 43-101.
9. I have worked on or been associated with the Property since 2007.
10. I have read NI 43-101 and the Sections of this report for which I am responsible have been prepared in compliance with the instrument.
11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Report Date: August 30, 2018
Effective Date of PEA: July 24, 2018
Effective Date of Updated Mineral Resource Estimates: May 7, 2018

"William Mercer" {signed and sealed as of report date}

William Mercer, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Donald H. Hains, P.Geo.

As the co-author of this report for Avalon Advanced Materials Inc. (Avalon) entitled "The East Kemptville Tin Production and Site Remediation Project Preliminary Economic Assessment, Nova Scotia, Canada", effective date July 24, 2018, I, Donald H. Hains, do hereby certify that:

1. I am President of Hains Engineering Company Limited, a company duly authorized by Professional Engineers of Ontario, with offices at 2275 Lakeshore Blvd. West, Suite 515, Toronto, ON M8V 3Y3.
2. I am a graduate of Queen's University, Kingston, ON with a degree in Chemistry (1974) and a graduate of Dalhousie University, Halifax, NS with an MBA (1976):
3. I am registered as a Professional Geoscientist in the Province of Ontario (Reg. # 0494). I have worked as a geologist and minerals economist for a since my graduation. My relevant experience for the purpose of the Technical Report includes:
   - NI 43-101 Technical report on MSFA Ltda tin deposit, Sao Francisco de Assis, Pará, Brazil (2014)
   - Due diligence review of Abu Dabbab tin-tantalum project, Egypt, 2011 and 2013
   - Due diligence review of alluvial tin/tantalite/columbite project in Sierra Leone (2013)
   - Due diligence reviews and technical report on tin/tantalite/lithium pegmatite deposits in Canada (1998, 2001, 2004, 2005, 2011, 2012), China (2004, 2005), Mozambique (2010, 2011), Zimbabwe (2000, 2001)
4. I visited the East Kemptville property on July 23-25 and September 3, 2014.
5. I am responsible for the Resource Estimate of the Low-Grade Stockpile and the text relevant to this resource estimate within the Technical Report.
6. I am independent of Avalon and related entities, as defined in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the property that is the subject of this Report.
8. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
9. To the best of my knowledge, information, and belief, this Report contains all scientific and technical information that is required to be disclosed to make the relevant section not misleading.

Report Date: XXX XX, 2018
Effective Date: July 24, 2018

"Donald H. Hains" {signed and sealed as of report date}

Donald H. Hains, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Reid Smith, M.A.Sc., P. Geo.

As the co-author of this report for Avalon Advanced Materials Inc. (Avalon) entitled "The East Kemptville Tin Production and Site Remediation Project Preliminary Economic Assessment, Nova Scotia, Canada", effective date July 24, 2018, I, Reid Smith, do hereby certify that:

1. I am the Team Leader, Environmental Services (Atlantic), Stantec Consulting Ltd., 102-40 Highfield Park Drive, Dartmouth, NS B3A OA3 e-mail: reid.smith@stantec.com
2. I hold the following academic qualifications:

   B.Sc. Acadia University (2004)

   Masters of Applied Science, Civil Engineering, Queen's University (2010)
3. I am a Qualified Professional (QP), and hold a Professional Geoscientist certification (NS, ON, NWT/NU)
4. I have worked in the minerals industry for over 7 years; my work experience has included consulting on environmental, hydrogeological and/or mine closure planning for proponents and/or regulators on projects such as: the Snap Lake, Gahcho Kue, Ekati and Diavik Diamond Projects (NWT), Mary River Iron Project (NU), Meadowbank Gold Project (NU), Touquoy Gold Project (NS) and GGM Hardrock Project (ON).
5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes environmental consulting and mine closure planning for several proposed and/or operating mining projects.
6. I have not visited the Property that is the subject of this report.
7. I am responsible for the section entitled "Environment and Socioeconomic Conditions", that provides a background of provincial and federal regulatory permitting approval processes, a baseline biophysical description and public and indigenous engagement, of Chapter 20 of this Technical Report.
8. I am independent of Avalon and related entities, as defined in Section 1.5 of NI 43-101.
9. I have worked on and been associated with the Property prior to this Technical Report.
10. I have read NI 43-101 and the Sections of this report for which I am responsible have been prepared in compliance with the instrument.
11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Report Date: August 30, 2018
Effective Date: July 24, 2018

"Reid Smith" {signed and sealed as of report date}

Reid Smith, M.A.Sc., P. Geo.