Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project Juab County, Utah

for

LITHIC RESOURCES LTD.

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1.0 SUMMARY

1.1 Introduction

The purpose of this report is to provide a technical summary of a Preliminary Economic Assessment (PEA) of the Crypto zinc project located in Juab County, Utah. The PEA is based on a resource estimate completed by Mine Development Associates in 2009 (Tietz et al, 2010) and sections 5-15, 17 and 18 of this report are repeated from their report. This report was written in compliance with disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1.

The Crypto zinc-copper-indium project is located on the northwestern edge of the Fish Springs Range in west-central Utah, approximately 160km southwest of Salt Lake City. The property consists of 174 unpatented lode mining claims; all or part interest in 18 patented mining claims, which are now private land; and one state mineral lease. The property has an aggregate area of 1,723.3ha and is situated in Sections 7-9 and 16-21, Township 11 South, Range 14 West, and in Sections 12, 13, and 24, Township 11 South, Range 15 West, Salt Lake Base and Meridian. All titles are held either in the name of Lithic or its U.S. subsidiary, N.P.R. (US), Inc.

The Crypto property is subject to a 1.5% Net Smelter Return royalty interest on all production payable to Vaaldiam Resources Ltd., which is also entitled to receive a one-time cash payment of $1,000,000 upon Lithic's securing financing to bring the Crypto property into production.

1.2 Geology and Mineralization

The Crypto zinc property is located on the northwestern pediment of the Fish Springs Range, one of the roughly north-trending ranges of the Basin and Range physiographic province. Generally speaking, the Fish Springs Range is a north-trending horst comprised of over 3,500m of Lower Cambrian to Upper Devonian platformal sedimentary rocks that have been homoclinally tilted to the west at generally moderate dips. Scattered quartz monzonite intrusive complexes and rhyolite dikes and occasional andesitic plugs intrude the sedimentary rocks locally and are thought to be Late Eocene in age. Tertiary volcanic rocks of latitic, shoshonitic, and rhyodacitic composition are found along with younger alluvium in basins adjoining the Fish Springs Range.

The Crypto property is predominately underlain by a sequence of lower Cambrian to upper Devonian dolostones with lesser, generally thin-bedded limestones and minor interbedded quartzites and shales. The most obvious structure on the property is the Juab fault, a west-northwest-trending, north-dipping normal fault that trends through the middle of the property. The fault is significant enough to have caused a conspicuous left-lateral offset of the Fish Springs Range and is thought to have on the order of 500-600m of net vertical displacement. The north-trending Overland fault along the western margin of the Fish Spring Range, together with a number of lesser subparallel faults, are normal with west-side down and represent extensional faulting typical of the Basin and Range Province.

Drilling has revealed that the sedimentary package has been intruded by a late Eocene (38.5 ±1.0 Ma, K/Ar), felsic intrusive complex which underlies a large part of the property at depth but is not exposed at the surface. In the vicinity of the Crypto zinc deposit, it rises to the bedrock surface in a cupola which sub-crops below shallow Pleistocene gravels. Various dikes ranging in composition from porphyritic quartz trachyte to rhyolite are exposed in outcrop or have been observed in drill core and are thought to be related to the same intrusive event as the quartz monzonite.

Thin-bedded carbonate rocks near the intrusive complex, particularly those with shaly partings, have been altered to skarn, marble, and siliceous hornfels. More thickly bedded to massive dolostones tend to be converted to marble. Zinc-copper-indium mineralization of the Crypto deposit occurs in portions of the skarn, while molybdenite mineralization occurs both as disseminations in the skarn and in more classic, porphyry-style quartz-pyrite veinlets within the quartz monzonite intrusion itself. High-grade silver-lead-zinc- mineralization, including that exploited in the historic Utah and Emma mines, occurs in structurally controlled replacement zones in carbonate rocks peripheral to skarn.

The most significant mineralization discovered to date on the Crypto property consists of sphalerite with minor chalcopyrite occurring in a series of concordant to discordant skarns and replacement bodies in carbonate rocks in proximity to the quartz monzonite intrusive complex. Two main areas of zinc mineralization have been found, the Main and Deep Zones, neither of which is exposed in outcrop. The two zones are separated by the Juab fault. The Main Zone occurs in Ordovician Pogonip Group carbonate rocks and possibly in some Ely Springs Dolomite immediately south of the sub-cropping quartz monzonite cupola and north of the Juab fault. Main Zone mineralization has been traced with drilling over a length of about 525m, a width of about 150m, and to a depth of 575m and remains open to the west and to depth. The Main Zone has been oxidized to an average depth of about 200m. The Deep Zone is hosted predominantly by thinly bedded, shaly limestones , probably the Corset Spring Shale and Candland Shale members, of the Orr Formation immediately south of the Juab fault. Deep Zone mineralization also occurs as proximal skarn within the Orr Formation along the contact with the underlying quartz monzonite intrusive. At least three separate mineralized stratabound horizons have been identified through drilling over an area of about 330m by 225m at depths of between about 450 to 750m. They remain open to the west, south, and possibly the east.

1.3 Exploration and Mining History

The Crypto zinc property lies in the Fish Springs mining district, from which about 7.9 million kg of lead, 1,300kg of zinc, 2,400kg of copper, and 2.7 million ounces of silver were produced from 1890 to 1953. This mineralization came from high-grade lead-silver ores found in oxidized carbonate replacement deposits.

The Crypto project area was at one time or another held or optioned by Kennecott Copper Corporation (“Kennecott”), Pinnacle Exploration Inc. (“Pinnacle”), Utah Construction & Mining (“Utah,” later Utah International Inc.), Noble Peak Resources Ltd. (“Noble Peak,” later called Vaaldiam Resources Ltd. “Vaaldiam”), Cyprus Minerals Company (“Cyprus” (in a joint venture with Mitsui Mining & Smelting Co. Ltd.)), Sierra Gigantes Resources Inc. (“Sierra”), and

EuroZinc Mining Corporation ("EuroZinc"). Lithic purchased the property from EuroZinc in 2005 through the purchase of N.P.R. (US), Inc., a wholly-owned subsidiary of Eurozinc, and subsequently staked an additional 119 claims, purchased additional interests in two of the patented claims, and purchased a partial interest in one additional patented claim. The Main Zone was discovered by Utah, and the Deep Zone was later confirmed as a separate zone and expanded by Cyprus. Historic resource estimates were made by Utah in 1976, Cyprus in 1991 and 1993, and Noble Peak in 1995, but all of these were made prior to the NI 43-101 reporting requirements.

Since acquiring the property, Lithic has completed aerial photography and photogrammetry, pole-dipole induced polarization surveying, and helicopter-borne magnetic surveying. In 2007-2008, Lithic completed a program of 10,639m of core drilling in 17 holes. In 2009, they commissioned preliminary metallurgical testwork on both sulfide and oxide mineralization.

1.4 Drilling and Sampling

Four campaigns of drilling completing 85 core and RC exploration holes totalling 38,138m have been carried out on the Crypto property (Table 1.1).

Table 1.1 Crypto Mineral Resource Drilling Database Summary

Company	Period	Hole Numbers	Core		RC		Total	
			No.	Meters	No.	Meters	No.	Meters
Pinnacle Mines	1958-1959	C1,C2	2	228.6	--	--	2	228.6
Utah International	1961-1979	CC1 to 46 RC2-4,14,26,35-37	39	16,555.8	8	609.5	47	17,165.3
Cyprus Minerals	1990-1991	CCC1 to15 RCCC-1, 6	17	9,434.6	2	670.6	19	10,105.2
Lithic Resources	2007-2008	C07-1 to 5 C08-6 to 15	17	10,638.9	--	--	17	10,638.9
Totals			**75**	**36,857.9**	**10**	**1,280.1**	**85**	**38,138.0**

Lithic's samples were analyzed by ALS Chemex. Lithic implemented a quality-control protocol involving a variety of standards and duplicates as well as a blank for all of its sampling on the project. Every 11th sample was either a standard, a duplicate of some type (preparation, assay, or field duplicate), or a blank.

MDA had audited the historic database using the available sources and had no reason to suspect that any systematic problems exist; however, one deficiency is the lack of original analytical certificates from previous exploration programs. As a result, MDA was not able to state that it had checked the historic database using entirely original sources. MDA found no significant issues in its audit of Lithic's entries in the database.

1.5 Mineral Processing and Metallurgical Testing

G&T Metallurgical Services Ltd. tested a series of sulfide composites constructed from samples across the deposit to develop a flotation treatment plan for sulfide mineralization. Kappes, Cassiday & Associates completed a series of leach and concentration tests on a composite of the oxide zinc mineralization constructed from samples taken from a variety of Lithic's drill intercepts.

For G&T's study of sulfide mineralization, composites were created representing three different types of mineralization: Cu-Zn, High-Grade Zn, and Low-Grade Zinc mineralization. The program developed by G&T focused on the conventional processes to generate independent copper and zinc concentrates. A Bond ball mill work index, grinding and flotation tests, open-circuit and locked-cycle tests, and preliminary testing to investigate potential to recover a magnetite concentrate were undertaken.

The initial testing on the Cu-Zn composite, along with subsequent testing on the Low-Grade Zn and High-Grade Zn composites, indicated a similar moderate hardness and the target grind was determined to be 65 microns. Energy requirements are expected to be similar for all three types of mineralization.

Open-circuit bench-scale testing to evaluate the recovery of copper and zinc determined that the use of zinc depressants, $ZnSO_4$ and NaCN, was essential to control the recovery of zinc to the copper concentrate thereby optimizing the performance of the zinc recovery circuit. The open-circuit testwork also indicated that rougher concentrate regrinds were essential for both the copper and zinc circuits in order to optimize the concentrate grades and recoveries.

The open-circuit testwork provided the preliminary criteria to develop flotation locked-cycle test procedures. Locked-cycle tests were completed to confirm recoveries and concentrate grades identified by the open-circuit tests. Among the results of the locked-cycle tests were:

- The zinc concentrate will contain 52.5% Zn which represents a zinc recovery between 75% and 95% depending on feed grade. The linear relationship between feed grade and recovery is based on the formula: Recovery Zn = (1.55 x %Zn(feed) + 78.43)%, max 95%.
- The copper concentrate will contain 32.0% Cu and 6.0% Zn which represents a copper recovery of 50% and a Zn recovery of 3%;
- Indium is recovered in the zinc concentrate described in the following formula: Recovery In = (-1.93 x %Zn(feed) + 76.8)%

The metallurgical testwork completed by G&T confirms that the sulfide zone of the Crypto project will be amenable to processing using a conventional copper – zinc differential flotation process. Gold, silver, and indium will be recovered to the concentrates. Deleterious elements were not found to be present in the copper and zinc flotation concentrates at penalty levels. However, it was recommended by G&T that future work should monitor bismuth concentration as it approached threshold limits depending on the smelter accepting the concentrate.

Kappes, Cassidy and Associates (KCA) completed a preliminary zinc mineral evaluation program on a single composite of the oxide mineralization. The preliminary leach tests indicated that copper, zinc, and indium can be extracted from the oxide material using a sulfuric acid leach process. The acid leaching test results indicated up to 95% of the zinc, 78% of the copper, and 37% of the indium were extracted from the ore. There was no work completed to determine the process method and resulting recovery of zinc, copper, or indium from the acid solution. There are proven technologies available that will be evaluated in future test programs.

The Preliminary Economic Assessment considered only sulfide ore as this was felt to have the best prospects for an economic situation. The oxide mineralization is considered to represent an opportunity for future evaluation, as is the possibility of generating a magnetite concentrate from the flotation tails.

A 3,500 tpd concentrator and a tailings facility along with offices and machine shops would be built at the site. Personnel would be bussed in from surrounding communities. An existing substation and single phase power line servicing the property would be upgraded to accommodate the increased power requirements of the mining and processing operation.

The process flowsheet for sulfide ore will include crushing and grinding facilities to generate a flotation feed with a nominal P80 of 65 microns. For ores with sufficient copper to operate an economic recovery process, zinc depressants will be added to the flotation feed slurry to minimize zinc recovery to the copper rougher concentrate. Copper rougher concentrate will be reground to 15 microns prior to three stages of copper cleaner flotation to produce a concentrate grading 32% Cu. For the purpose of the PEA, copper recovery is projected to be 80%.

Tailings from the copper flotation circuit will be fed to the zinc flotation circuit. $CuSO_4$ will be added to the slurry to activate the zinc (sphalerite) for flotation. Zinc rougher concentrate will be reground to 35 microns prior to three stages of zinc cleaner flotation to produce a concentrate grading 52.5% Zn at an average recovery of 83.6% over the mine life. Over the projected mine life, the indium grade of the zinc concentrate averages 246 g/t at a recovery of 58.2%. Concentrates from the flotation process will be thickened and filtered to provide dry concentrates that will be shipped to the respective smelters.

1.6 Mineral Resource Estimation

Upon completion of the database validation process, MDA constructed cross sections looking west. One set of sections was made for each of lithology, zinc, copper, and indium. Drill-hole information, including host lithology and metal grades along with the topographic surface, were plotted on the cross sections. MDA used a combination of lithology, angles to core axes, structural data and logged sulfide percentages to construct a geologic model which formed the basis for the density model and metal domain models. MDA assigned density values to various group of rocks ranging from a low of 2.54 to a high of 3.97 g/cm^3. The significant range in density values reflects the variable high-sulfide or high-magnetite alteration/mineralization.

Quantile plots of zinc and copper in percent and indium in grams per tonne were made to help define the natural populations of metal grades to be modeled on the cross sections. The plots were reviewed with all data for each metal grouped together, but also with the sulfide material evaluated separately from the oxide. The analytical population breaks indicated by the quantile plots along with the geological interpretation were used in the creation of distinct mineral domains. The interpreted cross sections were then sliced to levels on 3m intervals to coincide with the block-model block size in that direction. The defined metal mineral domains were used to code the drill samples and control the estimation. Mineral domain statistics, and spatial location of higher grades, were made to assess validity of these domains and to determine capping levels. After these analyses, MDA chose to cap 23 samples. Compositing was done to 3m down-hole lengths, honoring all material type and mineral domain boundaries. The 2m by 2m by 3m blocks inside each mineral domain were estimated using only composites from inside that domain.

The Crypto resource block model replicates the relatively complex metal distributions and skarn geometries. Because of the rather contorted geometries, two passes using Inverse Distance techniques were made in the estimate; a long pass to insure filling in all the blocks and a short pass for the Indicated classification. Indium search parameters were particularly long (up to 300m) because of the limited amount of analytical data.

MDA classified the Crypto resource in order of increasing geological and quantitative confidence into Inferred and Indicated categories to be in compliance with Canadian National Instrument 43-101 and the "CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines," issued in 2000 and modified with adoption of the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" in 2005. MDA classified the Crypto resources by a combination of distance to the nearest sample and the number of samples, while at the same time taking into account reliability of underlying data and understanding and use of the geology. There are no Measured resources within the deposit, primarily due to complexity of the mineralization but also due to limited drill data. The maximum distance criteria for Indicated within the Main Zone is less than that used for the Deep Zone due to the greater variability in domain morphology and metal grades. While MDA is confident that the indium zones do continue as modeled, mostly because of their relationship with zinc (the probable location of indium metal), MDA cannot be sure of the grades, and hence the unfortunately small amount of Indicated material. None of these issues deter from the overall confidence in the global project resource, but they do detract from confidence in some of the accuracy which MDA believes is required for Measured and Indicated resources. Without the downgrading for indium, the Indicated zinc and copper material would likely be two times larger than presently reported.

A summary of the total combined Crypto resources are tabulated in Table 1.2.

Table 1.2 Summary Table of Crypto Resources

Crypto Project Reported Resource

Class	Cutoff ZnEq%	Tonnes	ZnEq (%)	Zn (%)	Zn (tonnes)	Zn (lbs)	Cu (%)	Cu (tonnes)	Cu (lbs)	In (g/t)	In (kg)
Oxide - Indicated	1.00	1,114,000	5.48	4.54	50,536	111,413,000	0.263	2,925	6,449,000	10.3	11,500
Sulfide - Indicated	3.00	5,800,000	6.60	4.44	257,709	568,151,000	0.309	17,892	39,446,000	48.8	283,100
Total Indicated		6,914,000	6.42	4.46	308,245	679,564,000	0.301	20,817	45,895,000	42.6	294,600
Oxide - Inferred	1.00	4,644,000	4.45	3.73	173,414	382,312,000	0.165	7,680	16,932,000	12.5	58,300
Sulfide - Inferred	3.00	13,805,000	6.83	4.84	667,714	1,472,057,000	0.372	51,342	113,191,000	37.4	516,400
Total Inferred		18,449,000	6.23	4.56	841,128	1,854,369,000	0.320	59,022	130,123,000	31.2	574,700

The stated resources are fully diluted to 2m by 2m by 3m (vertical) blocks and are tabulated on a zinc-equivalent ("ZnEq") cutoff grade of 1.0% for oxide material expected to be mined by open-pit methods and 3.0% for sulfide material to be mined using underground methods.. All material, regardless of which metal is present and which is absent, is tabulated. Because multiple metals exist, but do not on a local scale necessarily co-exist, the ZnEq grade is used for tabulation. Using the individual metal grades of each block, the ZnEq grade is calculated using the following formula:

$$\%ZnEq = \%Zn + (0.0284 * ppm\ In) + (2.5 * \%Cu)$$

This formula is based on prices of US$0.80 per pound zinc, US$2.00 per pound copper, and US$500.00 per kilogram of indium. No metal recoveries are applied, as this is the *in situ* resource.

The decision as to the appropriate mining method for the Crypto deposit awaits further evaluation, and there is the potential that a combination of open-pit and underground methods would be optimal. The lower cut-off used for the oxide material reflects the potential for open-pit mining scenarios for this near-surface material. The 2m by 2m by 3m block size likely understates the dilution expected from standard open-pit mining methods, but this block size was used to provide the operator the option for evaluating the deposit, either in total or within specific areas, using underground mining methods.

The detailed work completed by Lithic and MDA on the geologic model, and the data defining the model, resulted in a resource estimate of high quality. The risk is mostly related to the deposit type. Skarn deposits, such as Crypto, often have relatively complicated and rapidly changing grades and geology. This downside would mostly be alleviated through additional drilling. The upside is clearly dominated by the ability to increase the amount of higher classification material with an increase in indium sample assays. The relatively small amount of Indicated material within the current resource is due to the many fewer samples with indium grades. An increase in indium sample assays would likely increase the overall indium grade if the tenor of the new sample assays was similar to the existing assays. There is also good potential to increase the size of the deposit by targeting extensions of mineralization primarily to the east, west and south.

1.7 Underground Mining

The PEA considers the mining of sulfide mineralization between elevations 1,220 meters and 710 meters. Mineralized zones above the mining cut off grade extend over strike lengths of up to about 600 meters and occur within a horizontal width of between about 200 and 300 meters. In this area, the mineable zones generally dip greater than 60º and vary in horizontal thickness up to about 15 m.

The total mineral resource considered for underground mining is 12.92 Mt at average grades of 5.97% Zn, 0.44% Cu and 40.2 g/t In. After applying extraction, dilution and recovery factors, the total mineable material is 12.14 Mt at average grades of 5.19% Zn, 0.38% Cu and 35.0 g/t In.

Based on the geometry of the mineable zones, the use of longitudinal retreat stoping with uncemented rock fill is assumed for all areas. A preliminary comparison of mine development using shaft hoisting versus ramp access indicated that a better economic result was provided by the ramp development option. Primary access will be via two ramp systems developed from surface. Bored raises are used as ventilation airways and as passes for stope rock fill. Two active production horizons and the strike length of the mineable material of up to about 600 m will provide multiple mining areas.

The preproduction development and construction period is expected to be two years following two years of exploration and feasibility studies. Portal construction for the two declines is scheduled to start at the same time. The 1130 m level is reached in the third quarter of project year two. The 1010 m level is reached in the first quarter of year three and stope development on that level is scheduled to start in the second half of the year. Stope development starts on the second production horizon, on 830 m level, in year five.

Ore production in the fourth quarter of year three is forecast to average about 1,000 t/d. As more stopes are developed the production rate is forecast to increase to the design rate of 3,500 t/d by the middle of year four. This rate is sustained until the end of year ten and then decreases until mining is completed late in year eleven.

1.8 Tailings Storage Facility

A tailings storage facility (TSF) was located immediately west of the deposit and was designed to provide containment for 12.1 million tonnes of tailings solids at an assumed dry density of 1.4 tonnes/m^3. It covers an approximate area of 1.1 Mm^2 and includes a low permeability basin liner/subgrade material with a basin underdrain for seepage control.

The tailings were assumed to be conventional slurry tailings with a solids content of 30% as per discussions with Lithic. A small supernatant pond was assumed to be located on the western side of the facility. The starter dam is sized for containment of two years of operations at full mill throughput. The project site is located in an arid environment and there will be little water available to the process from local surface runoff. Some water may be available from a well field but an allowance has been included in the cost estimate for make-up water which may involve sourcing water from across the valley to the west.

The make-up water requirements for the mill can be significantly reduced by thickening the tailings at the mill and capturing the thickener overflow rather than discharging the water to the TSF where it evaporates. There are other potential options for optimizing the pipeworks and deposition systems to accommodate thickened tailings and these should be considered as potential opportunities to consider in future studies.

The TSF will be required to maintain long-term physical and geochemical stability, protect the downstream environment, and manage surface water. The preliminary closure and reclamation plan for the TSF includes constructing a TSF cover at closure and construction of an emergency overflow spillway.

1.9 Economic Analysis

The economic assessment in this report is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that this Preliminary Economic Assessment will be realized.

In the early stages of this study, it was determined that a minimum processing rate of 3,500 tpd was necessary to maximize economies of scale. As a result, the model considered all sulfide resources, including lower grade mineralization, and used a zinc price of $1.10 per pound to outline sufficient tonnage above a cutoff grade of 4% zinc equivalent in order to support a minimum ten year mine life. Other metal price assumptions are listed in Table 1.3.

Table 1.3 Metal Price Assumptions

Metal Prices		US $
Zinc	Per lb	$1.10
Copper	Per lb	$2.00
Indium	Per kg	$500.00
Gold	Per oz	$850.00
Silver	Per oz	$12.00

Although gold and silver were not included in the resource model, metallurgical testwork has shown that substantial quantities of these elements are present in the copper concentrate. They have been included in the cash flow analysis at concentrate grades of 21 g/t Au and 3,000 g/t Ag.

Operating and capital costs along with other key parameters are summarized in Table 1.4.

Table 1.4 Summary Cost and Production Parameters

Pre-production capital	US$ millions	186.9
Sustaining capital	"	118.9
Total Capital	"	305.8
Mining/processing rate	tonnes per day	3,500
Mine life	years	11
Operating costs		
mining	US$/tonne	32.36
processing	"	21.66
tailings water management	"	0.25
G&A	"	10.00
Total operating cost	"	64.28
Average Annual Production		
zinc	lb	89,285,000
copper	lb	7,138,000
indium	kg	21,000
gold	oz	7,000
silver	oz	1,009,000

The Net Present Value (NPV) for the project has been estimated using discounted cash flow methods for a range of discount rates as summarized in Table 19-15.

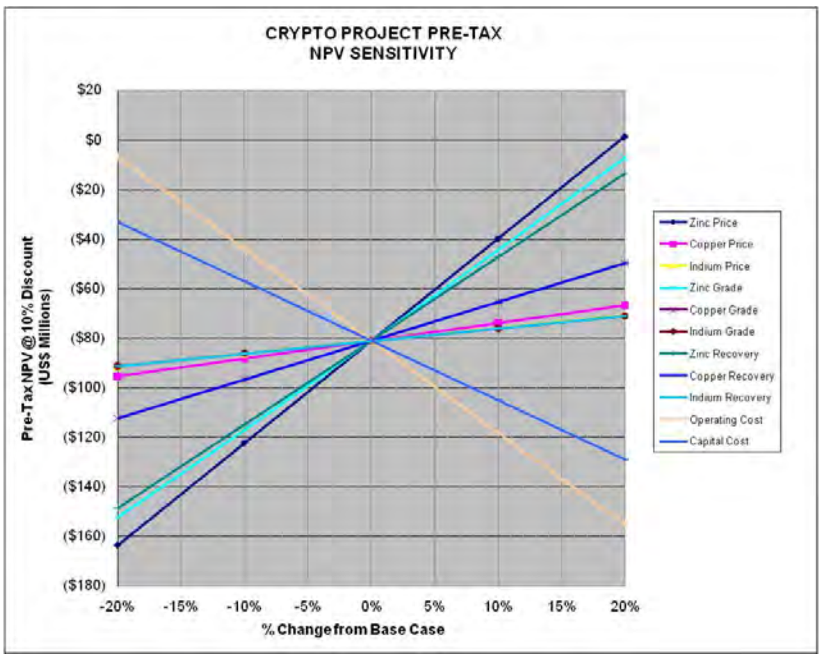
Table 1.5 Pre-Tax NPV at Various Discount Rates

Discount Rate	NPV (US$)
2%	$7,752,289
4%	($22,694,300)
5%	($35,397,036)
6%	($46,668,188)
8%	($65,531,419)
10%	($80,340,706)
12%	($91,920,604)
15%	($104,612,224)

At a 10% discount rate, the project pre-tax NPV from beginning of the development period is -$80.3 million. The internal rate of return is 2.5%.

Figure 19-9 illustrates the results of a sensitivity analysis of key economic inputs to the Crypto cash flow model. The most sensitive parameters in terms of equivalent percent change are zinc grade, zinc recovery, zinc price, operating cost and capital cost.

Figure 1.1 Pre-tax NPV Sensitivity



The project has a payback period equal to the mine life. A grade increase of 30% to 40% is required to demonstrate reasonable economic results for a projected mine life of ten years at a processing rate of 3,500 tpd. Increasing the cutoff grade of the existing resources to the point at which the average zinc grade is increased by 30% results in a loss of approximately 40% to 45%

of resources available for mine scheduling. Further exploration is required to locate additional relatively higher grade resources that could increase the overall grade of the mineable resource and which might also be available for early mine scheduling. Existing lower grade resources could be used to extend mine life after return on investment has been achieved.

1.10 Conclusions and Recommendations

Substantial Indicated and Inferred resources of skarn-hosted zinc, copper, and indium have been outlined on the Crypto property. While their overall grade is too low to be economic at this point, the resources outlined to date contain areas with higher grades. In aggregate, these represent approximately 55-60% of the minimum tonnage and grade required for an economic situation. Since there is good potential to expand existing resources to the east, west and south, and there is very good potential for the discovery of new zones beyond these extensions, it is entirely possible that enough additional mineralization at sufficient grade will be found to improve the potential economics of the project.

Further drilling is clearly warranted to explore for additional, relatively higher grade mineralization that could increase the overall grade of the mineable resource and which might also be available for early mine scheduling. Existing lower grade resources could be used to extend mine life after payback of development capital has been achieved.

The next phase of work at Crypto should target expansion of higher grade resources through approximately 10,000 meters of core drilling with a budget of approximately US$3.6 million.

2.0 INTRODUCTION AND TERMS OF REFERENCE

This technical report on the Crypto zinc project, located in Juab County, Utah, was prepared at the request of Lithic Resources Ltd. ("Lithic"). Lithic is a public company based in Vancouver, Canada, which trades on the TSX Venture Exchange. The Crypto property is held by Lithic and its U. S. subsidiary, N.P.R. (US), Inc. ("NPR").

This report was written in compliance with disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1.

2.1 Project Scope and Terms of Reference

The purpose of this report is to provide a technical summary of a Preliminary Economic Assessment of the Crypto zinc project based on a resource estimate by Mine Development Associates (Tietz et al, 2010) ("MDA"). The report restates, verbatim, sections 5-15, 17 and 18 from Tietz et al (2010) and includes updated or new sections as follows:

Section	Title	Author
16	Mineral Processing and Metallurgical Testing	Ken Major, P.Eng.
19.1	Underground Mining	Keith Durston, P.Eng.
19.2	Tailings Disposal	Knight Piésold Ltd.
19.3	Economic Analysis	John Nilsson, P.Eng.
20	Interpretation and Conclusions	Nilsson, Major, Durston
21	Recommendations	Nilsson, Major, Durston

There is no affiliation between Mssrs. Major, Durston, Nilsson, Ristorcelli or Tietz, or Knight Piésold, and Lithic except that of an independent consultant/client relationship.

The scope of this study included a review of pertinent technical reports and data provided to authors by Lithic relative to the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, drilling programs, and metallurgy. The authors have relied on the data and information provided by Lithic for the completion of this report.

The authors' mandate was to comment on substantive public or private documents and technical information listed in the References. The mandate also required on-site inspections and the preparation of this independent technical report containing the authors' observations, conclusions, and recommendations. Tietz, accompanied by Peter Ronning (consultant acting on MDA's behalf), conducted a site visit on March 26, 2008, which included a review of the drilling and sampling procedures. Ristorcelli conducted a site visit on April 6, 2009. Tietz conducted a second site visit June 9 through 13, 2008 in which the drilling results and project geology were reviewed with Lithic personnel.

The authors have made such independent investigations as deemed necessary in their professional judgment to be able to reasonably present the conclusions discussed herein.

2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

In this report, measurements are generally reported in metric units. Where information was originally reported in Imperial units, MDA has made conversions according to the formulas shown below; discrepancies may result in slight variations from the original data in some cases.

Linear Measure

1 centimeter	= 0.3937 inch	
1 meter	= 3.2808 feet	= 1.0936 yard
1 kilometer	= 0.6214 mile	

Area Measure

1 hectare	= 2.471 acres	= 0.0039 square mile

Capacity Measure (liquid)

1 liter	= 0.2642 US gallons

Weight

1 tonne	= 1.1023 short tons	= 2,205 pounds
1 kilogram	= 2.205 pounds	

Currency Unless otherwise indicated, all references to dollars ($) in this report refer to currency of the United States.

<u>Acronyms and abbreviations that appear in report:</u>

AA	atomic absorption spectrometer
Ag	silver
ASL	above sea level
Cd	cadmium
CIM	Canadian Institute of Mining, Metallurgical, and Petroleum
cm	centimeter
core	diamond drill drilling method
Cu	copper
^{o}C	degrees Centigrade
dmt	dry metric tonne
ddh	diamond drill holes
FA-AA	fire assay with an atomic absorption finish
Fe	iron
GSR	gross smelter return
g	grams
g Ag/t	grams of silver per metric tonne
ha	hectares
ICP-AES	inductively coupled plasma-atomic emission spectroscopy analytical technique
ICP-MS	inductively coupled plasma mass spectroscopy analytical technique
In	indium

IP	induced polarization geophysical survey
km	kilometers
m	meters
Ma	million years
mm	millimeter
NPI	net profits interest
NSR	net smelter return
oz	ounces
Pb	lead
QA/QC	quality control/quality assurance
RC	reverse-circulation drilling
t	tonnes
t/a	tonnes per annum
tpd	tonnes per day
Zn	zinc

3.0 RELIANCE ON OTHER EXPERTS

The authors are not experts in legal matters, such as the assessment of the legal validity of mining claims, private lands, mineral rights, and property agreements. The authors did not conduct any investigations of the environmental or social-economic issues associated with the Crypto project, and the authors are not experts with respect to these issues.

Information concerning land status as well as current legal title, information on the State of Utah mineral leases, and material environmental and permitting information that pertain to the Crypto property are provided in an independent report prepared for Lithic by North American Exploration (Gatten, 2010) Material terms of all agreements, existence of applicable royalty obligations, and other encumbrances are publically disclosed by Lithic on SEDAR. The authors offer no professional opinions with respect to the provided information.

4.0 PROPERTY DESCRIPTION AND LOCATION

The authors are not experts in land, legal, environmental, and permitting matters. The land-status information in Section 4.2, the State of Utah mineral lease information in Section 4.3, and the permitting/environmental information in Section 4.4 came primarily from a report on land status and mineral rights associated with the Crypto property prepared by North American Exploration, Inc. ("NAE") (Gatten, 2010); Mr. Gatten is a licensed professional geologist in the state of Utah with significant experience in land and permitting issues. Further clarification of the environmental information in Section 4.4.2 came from a written communication from North American Exploration.

4.1 Location

The Crypto zinc property is located in western Juab County, west-central Utah, U.S.A, approximately 160km southwest of Salt Lake City (Figure 4.1).

The property is located within the Fish Springs SW 7½' quadrangle and the Fish Springs 30 by 60 minute topographic map. The project area lies about 4km south of the U. S. Air Force's Deseret Test Center. The Fish Springs Wilderness Study Area is adjacent to the southern boundary and southeast corner of the Crypto property (Figure 4.2). Gatten (2010) noted that the Fish Springs National Wildlife Refuge is located about 5 to 6km east of the Crypto property on the opposite side of the range in an entirely different watershed and groundwater resource area.

The Crypto zinc project lies within the Fish Springs mining district.

Figure 4.1 Location of the Crypto Zinc Property

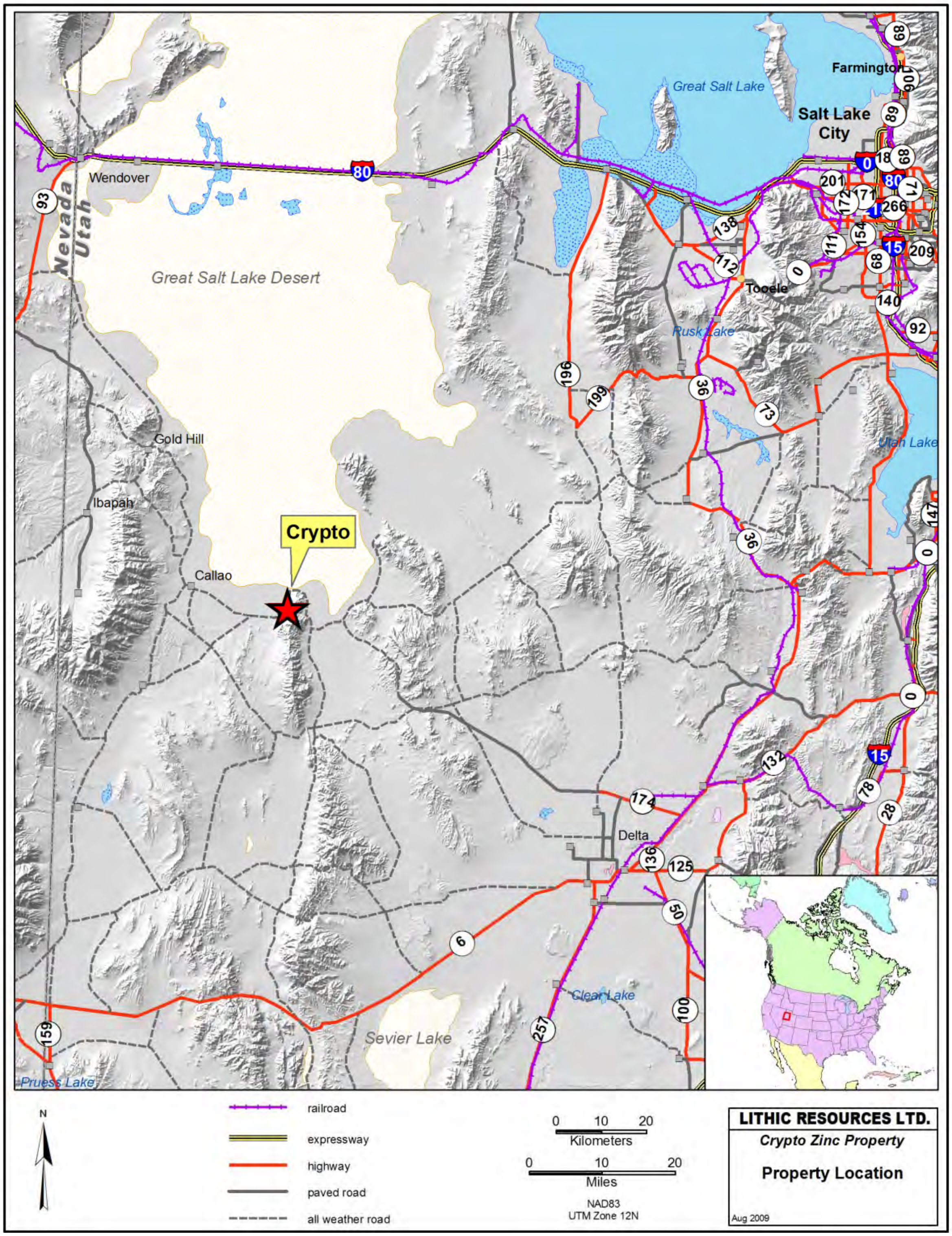
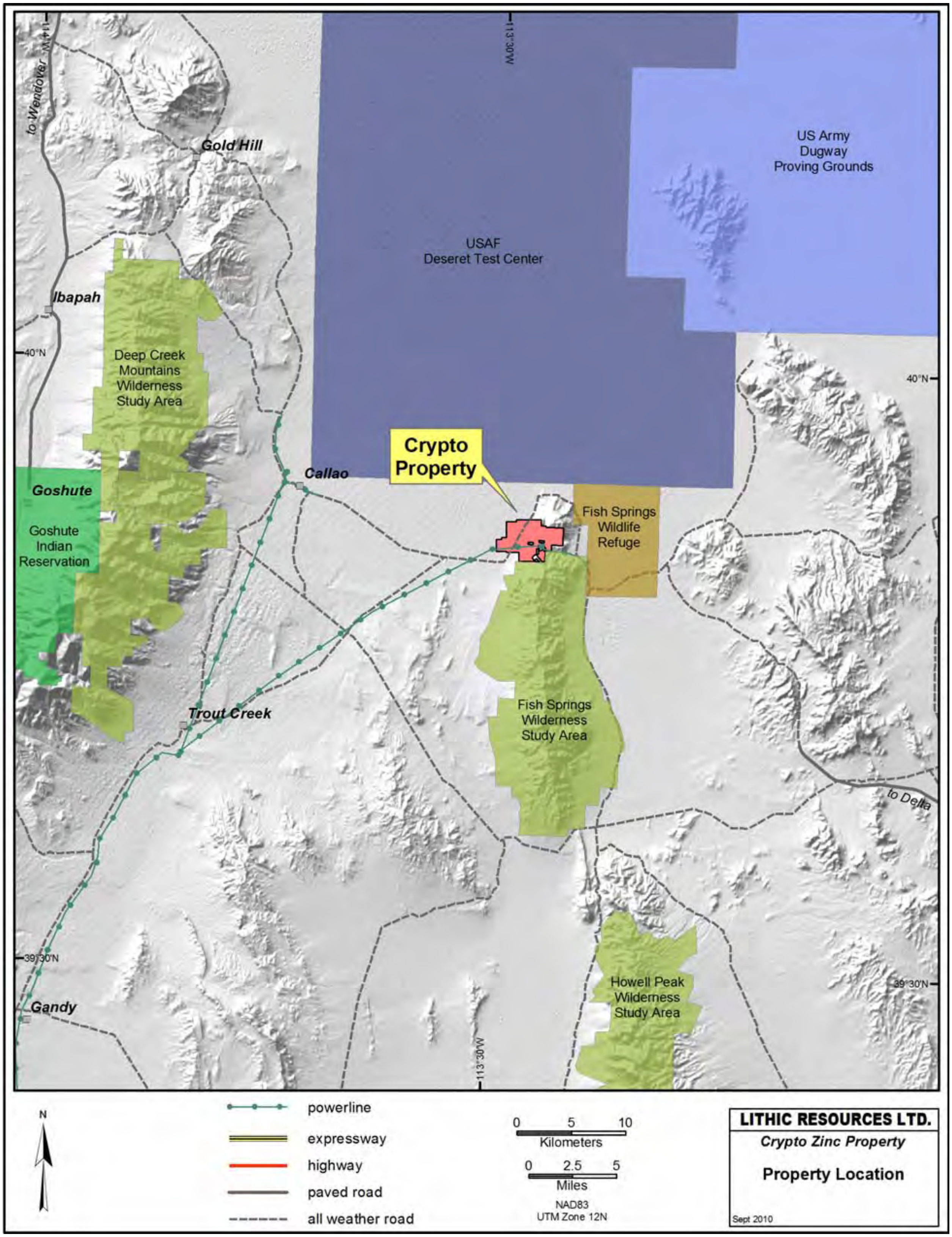


Figure 4.2 Land Status in the Vicinity of the Crypto Property



4.2 Land Area

The Crypto property consists of a single contiguous group of unpatented and patented lode mining claims and one Utah state mineral lease (Figure 4.3; Appendix A) situated in Sections 7-9 and 16-21, Township 11 South, Range 14 West, and in Sections 12, 13, and 24, Township 11 South, Range 15 West, Salt Lake Base and Meridian. All titles are held either in the name of Lithic or NPR. There are no records available to suggest that the property boundary and claims have been surveyed historically. The property comprises 174 unpatented mining claims, all or part interest in 18 patented mining claims (private land), and one state mineral lease with an aggregate area of 1,723.3ha.

The 174 unpatented claims, which comprise 1,352.1ha, are located on public land administered by the U.S. Bureau of Land Management ("BLM"). They are the Crypto Zn, Crypto, and Pony claims. Unpatented claims can be held indefinitely and require an annual maintenance fee of US$140 to be paid to the BLM each year on September 1 in order to remain in good standing. Gatten (2010) reported that the annual maintenance fees for the 174 unpatented claims have been paid for the 2010-2011 year, that the claims are valid until September 1, 2011, and that the claims are considered "active" and in good standing by the BLM. Lithic purchased 40 of the unpatented claims as part of the original land package they acquired and staked the remaining 134 claims themselves.

The 18 patented claims, which include both surface and mineral rights, are now considered private or fee land and as such are administered by the state of Utah. These 18 patented claims comprise 124ha in the core of the historic mining district. Through NPR, who is the owner of record, Lithic has 100% interest in 13 of the patented claims and a part interest in the five additional patented claims (Appendix A) (Gatten, 2010). According to Lithic, the patented claims may require varying but generally nominal annual property tax payments due in November of each year. Lithic represents that the annual property tax payments for the patented claims have been paid for the current year.

The Utah state mineral lease ML 48312, which comprises 247ha, is administered by the state of Utah School and Institutional Trust Lands Administration ("SITLA"). NPR entered into the lease June 16, 1999 (Gatten, 2009). Leases are typically for 10 years initially, during which time they require an annual payment of US$1 per acre due on July 1 of each year to remain in good standing. Following the initial 10-year period, the leaseholder may apply to extend the lease for another 10-year period based on evidence that significant work has been done to develop the property. Extended leases are subject to minimum advance royalties of US$3 per acre per year in addition to the annual lease payment of US$1 per acre per year. Mineral production or processing on state leases is subject to payments of royalties as summarized in Section 4.3. The state lease that is part of the Crypto property was due to expire July 1, 2009, but Lithic applied for and was granted an extension for an additional 10 years. The new lease requires advance royalty payments of $3 per acre per year in addition to the standard fee of $1 per acre per year; the total annual cost to hold this lease is now $2,448. Gatten (2010) reported that "*all payments are current and the lease is active. The terms of the lease extend to June 30, 2019, but can be extended further if a mine is in production.*"

Gatten (2010) opined that NAE's investigation "…indicates that the property is as represented and that Lithic Resources, Ltd. and NPR (US), Inc. is the owner. The unpatented mining claims are valid and in good order...The patented mining claims are owned by NPR (US), Inc. and the State Metalliferous Minerals Lease is also held by NPR (US), Inc."

The holding costs of the Crypto property in 2010 are estimated by Lithic at about US$27,000.

Within the Crypto property boundary, there are 11 full and partial claims held by other parties as shown on Figure 4.3.

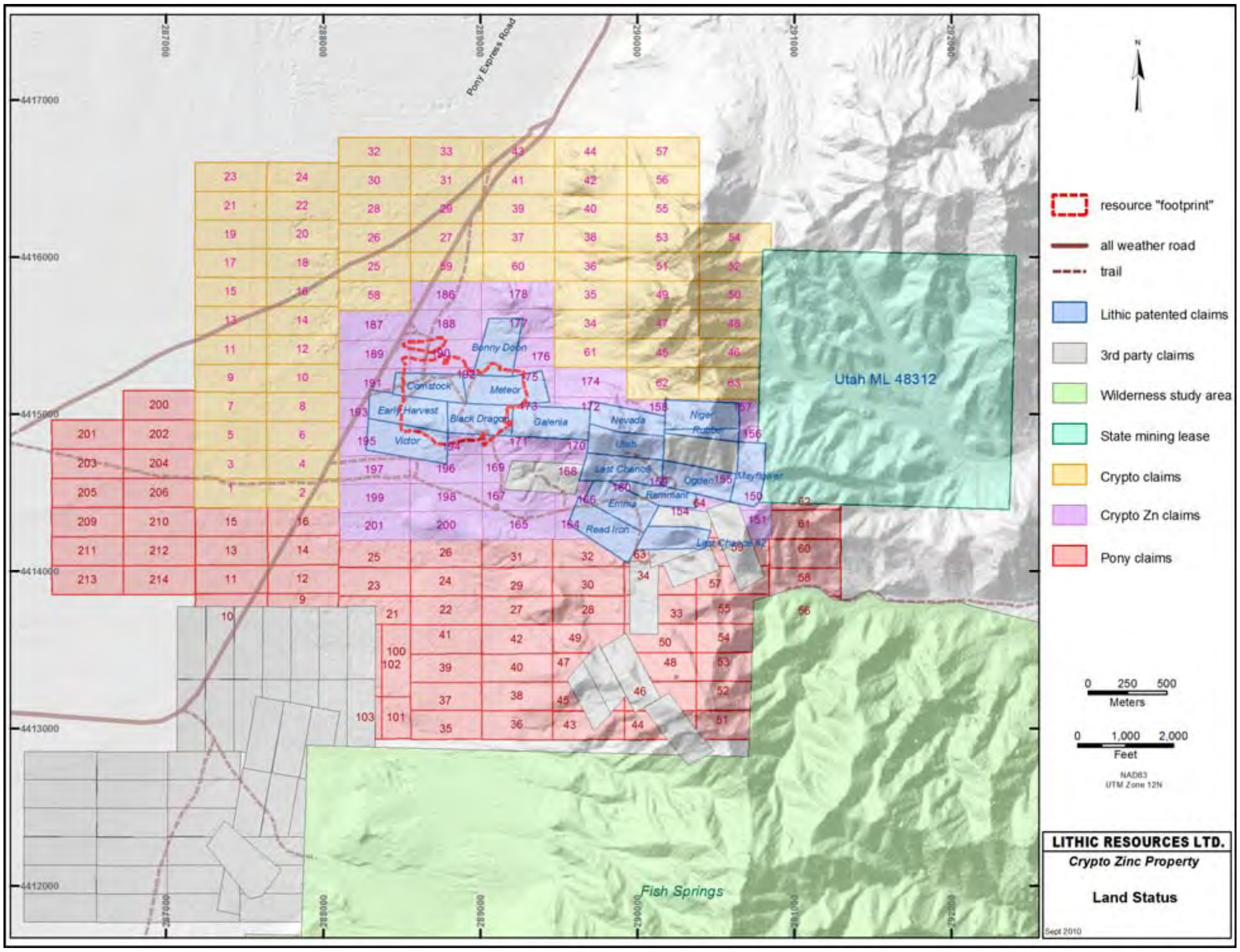
4.3 Agreements and Encumbrances

The information in this section has been provided by Lithic, through a corporate press release date July 21, 2005 currently found on SEDAR, and from Gatten (2009).

The Crypto property is subject to a 1.5% Net Smelter Return ("NSR") royalty interest on all production payable to Vaaldiam Resources Ltd. ("Vaaldiam"), the successor company to Noble Peak Resources Ltd., a previous owner of the property. Vaaldiam is also entitled to receive a one-time cash payment of $1,000,000 upon Lithic's securing financing to bring the Crypto property into production. These obligations were inherited through Lithic's purchase of the property in 2005.

The state mineral lease carries a Gross Smelter Return ("GSR") royalty of 8% for fissionable metalliferous materials and 4% for non-fissionable metalliferous minerals, payable to SITLA and based on any ores produced from the leased lands and sold by the lessee (Gatten, 2009). None of the current resource is contained within the leased state lands.

Figure 4.3 Property Map for the Crypto Project



4.4 Environmental Permits and Potential Liabilities

4.4.1 Permits

The following information on the status of permits is taken directly from the land report by Gatten (2010) with minor editorial changes to conform to the style of this report and with a summary table and figure added by Lithic.

Certain permits are required by the State of Utah to explore for minerals and conduct mining operations. These are administered by the Utah Division of Oil, Gas and Mining ("DOGM") and the BLM and include exploration, small mine (less than five acres disturbance), and large mine permits. In addition to DOGM regulation, the BLM also monitors field and reclamation activities conducted on federal lands.

NPR currently holds three permits in regards to the exploration and mine development of the Crypto property. These permits are:

Exploration Permit E/023/0105. This permit was first approved by DOGM on July 12, 2007 and has been renewed twice to allow for extensive drilling programs that have been conducted and/or contemplated on the property. The permit is in good standing until December 2012 and a reclamation contract is in place which includes a surety amount of $51,370. Annual permit fees have been paid through June 2011.

Small Mine Permit S/023/0103. This permit was first approved by DOGM April 6, 2009 and covers 5.0 acres of land located in the NW/NW and NE/SE of Section 18. The purpose of the permit is "for surface disturbances prior to pit development." The permit is in good standing until April 16, 2012 and a reclamation contract is in place which includes a surety amount of $23,280. Annual permit fees have been paid through June 2011.

Surface Management Notice UTU-079884. This permit was issued by the BLM and allows a total of 1.6 acres of disturbance consisting of drill pads and new roads supporting drilling operations within an area located on the border of Sections 17 and 18. The permit was issued 12/20/07 and has been renewed since then. It is in good standing until April 16, 2012.

The particulars of the permits are listed in Table 4.1 and illustrated on Figure 4.4.

Table 4.1 Summary of Permits in Effect on the Crypto Project

Permit	Land Status	Approval Date	Bond Amount	Authorized Disturbance	In Good Standing Until
Notice of Intention/Exploration Permit E/023/0105	Private Surface and Mineral rights	7/12/07; renewed 2/3/09	US$51,370	5 acres	12/2012
Small Mine Permit S/023/0103	Private Surface and Mineral rights	4/6/09	US$23,280	5 acres in Section 18	4/2012
Surface Management Notice UTU-079884	Public Surface rights, Mining Claims over Public Minerals rights	12/20/07	Joint with E/023/0105	1.6 acres in Sections 17-18	4/2012

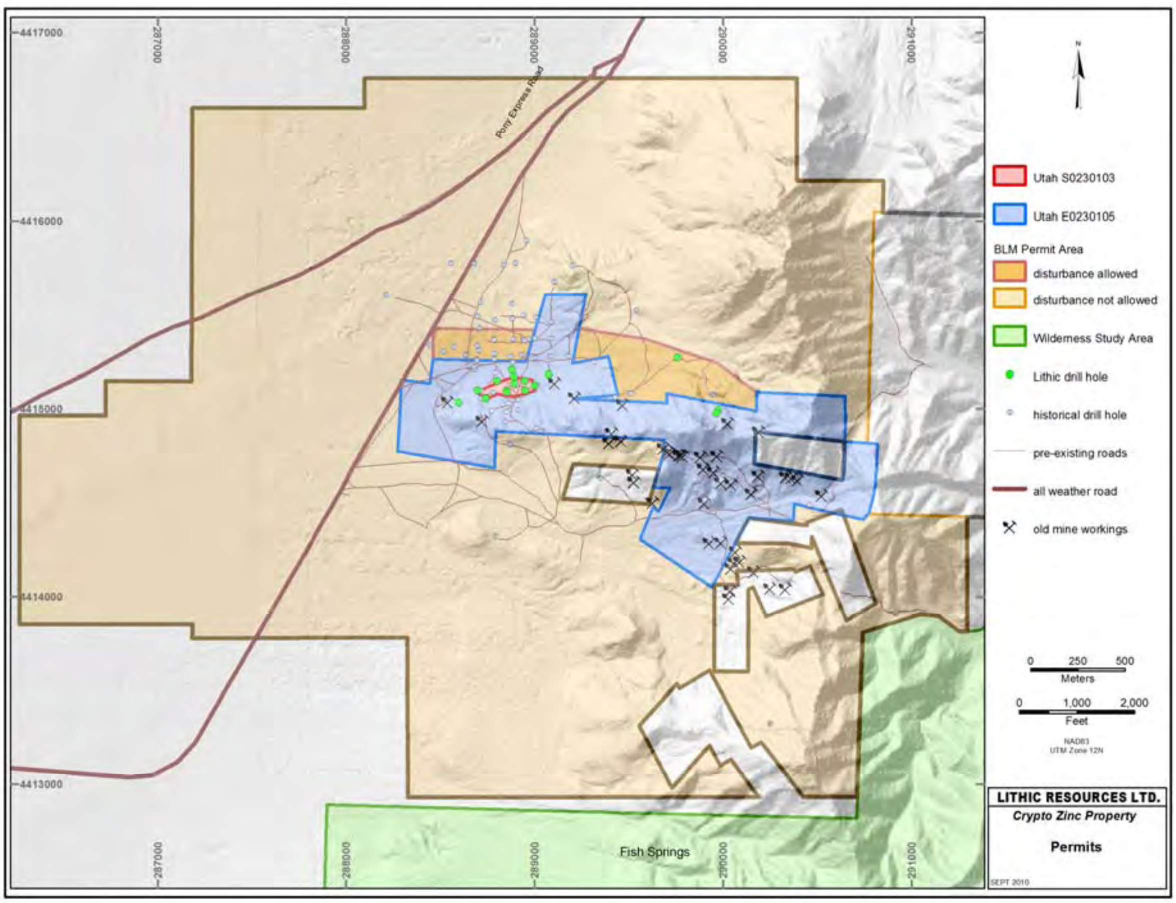
4.4.2 Potential Liabilities

According to Gatten (2010) and further written communication from Gatten, environmental liabilities on the Crypto property are limited to the reclamation of areas disturbed by Lithic's exploration activities. A long history of mining and exploration on and around the property prior to Lithic's purchase of the property in 2005 is evidenced by the presence of numerous mine workings including shafts, adits, and pits as well as waste dumps and debris from historic mining activities and settlements associated with those activities. Responsibilities in such areas of historical mining are governed by The Utah Code under Title 40 "Mines and Mining", and Chapter 8 the "Utah Mined Land Reclamation Act", which was enacted in 1975. Section 4 of Chapter 8 states that:

"lands not subject to reclamation ("Lands Affected") by a current Operator are defined in (13) (b) as "all lands shall be excluded .....that would (i) "be includeable as land affected, but which have been reclaimed in accordance with an approved plan, as may be approved by the board" and (ii) "lands in which mining operations have ceased prior to July 1, 1977".

The last recorded mining activity on the Crypto property dates from the 1950s. Lithic's exploration activities have avoided and will continue to avoid or mitigate any impacts in accordance with applicable permits and regulations.

Figure 4.4 Location of Permits in Effect on the Crypto Project



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

5.1 Access

The Crypto property is located approximately 160km southwest of Salt Lake City and can be accessed from that direction via state route 36, from which the Pony Express Road and other county-maintained gravel roads lead to the property. The property is also located 125km northwest of Delta, Utah, and can be accessed from that direction via the Brush Wellman Road and other county-maintained gravel roads. Numerous old drill and mining roads provide excellent access to most of the property.

5.2 Climate

Climatic conditions on the property are those of the arid desert, typical of the Great Basin area in Nevada and Utah. Average daytime temperatures range from 33ºC in July to 1ºC in January, with pronounced temperature variations between night and day. In general, precipitation does not exceed 15cm per year and occurs mainly as afternoon cloudbursts during the summer and as snowfall between October and May. Surface exploration can be carried out year-round on the property, although winter storms can occasionally deposit enough snow to impede access for a few days at a time. Mining could be carried out year-round.

5.3 Local Resources and Infrastructure

The nearest significant supply point is Delta, located approximately 125 road-km southeast of the property, which has a population of about 3,200. The state capitol and major metropolitan area of Salt Lake City is about 160km northeast of the property. Lithic established a small trailer camp on the property during its exploration program, as did previous property owners. Various ranchers in the area have heavy equipment such as bulldozers and excavators available for contract work, and unskilled labor is available in local settlements.

The nearest rail line runs through Delta; another runs parallel to Interstate Highway 80 north of the property and can be accessed at the town of Wendover, Utah, approximately 120km north-northwest of the property. The closest landing strip with the capacity to land small-engine fixed-wing aircraft is at the small ranching community of Granite Ranch, approximately 25km southwest of the property.

A single-phase 14kV electrical transmission line owned by the Mt. Wheeler Power cooperative in Ely, Nevada, crosses the Crypto property from the west, and three-phase electrical power is available at the ranching community of Callao about 20km to the west. A 1.5 gW coal-fired generating station owned by Intermountain Power is located immediately northwest of Delta and supplies Mt. Wheeler Power through a 230kV line passing south of the property at a distance of about 70km.

There is no surface water on the property. An old water well established in the 1960s was used for exploration drill water at the time. Subsequent exploration campaigns either supplemented this source with or relied on water purchased from landowners in the Callao and Granite Ranch settlements across the Snake Valley and trucked to the site.

5.4 Physiography

The Crypto property lies on the southern margin of the Great Salt Lake Desert and on the northwestern pediment slope of the Fish Springs Range, one of the generally north-trending mountain ranges of the Basin and Range physiographic province which covers western Utah and most of the state of Nevada. The Fish Springs Range is about 5km wide by 30km long and is 1,830m in elevation at its crest. The Crypto property includes flat valley floor at an elevation of about 1,200m on the west side of the property, which slopes up to fairly rugged cliffs and ridges with a maximum elevation of about 1,800m on the eastern side of the property. To date, work has concentrated on and around a minor east-west-trending spur extending west from the main range, at elevations of between 1,300 and 1,500m above sea level ("ASL").

Vegetation is typified by scattered low brush and grassy patches with intervening areas of bare ground. Mainly desert species are dominated by various grasses, sagebrush, greasewood, rabbit brush, shadscale, blackbrush, mormon tea, leadbush, and prickly pear cactus. The principal bird and animal species observed in the property area include various songbirds, rodents, jackrabbits, lizards, and snakes, while coyotes, deer, owls, and raptors are reported to have been seen elsewhere in the Fish Springs Range. Local ranchers occasionally run cattle in the area seasonally, but a lack of water precludes maintaining any livestock on a continuing basis.

Figure 5.1 shows the northern end of the Fish Springs Range. The Crypto resource is located at the foot of the mountains within the center of the image.

Figure 5.1 View Looking East toward the Crypto Property

6.0 HISTORY

6.1 Exploration History

The following information has been taken from a summary of the project's history apparently compiled by staff of Utah International in late 1973, from information provided by Lithic, and from additional sources as cited.

Silver-lead mineralization was discovered in the area of the present day Crypto zinc property in 1890 by C. C. Van Alstine, and the Fish Springs mining district was organized the following year (U.S.G.S. Mineral Resource Data System). Between 1890 and 1953, high-grade lead-silver ores were mined from oxidized, carbonate replacement deposits in small underground operations, mainly the Utah and Galena mines and to a much lesser extent, the Emma, Vulcan, Last Chance, and Meteor mines. The total production from the Fish Springs district between 1890 and 1953 is recorded at 7,850,929kg of lead, 1,271kg of zinc, 2,436kg of copper, 508 ounces of gold, and 2,658,220 ounces of silver from 18,415 tonnes of ore (Perry and McCarthy, 1976).

In 1953, Professor M. L. Jensen of Yale University, working for Kennecott Copper Corporation ("Kennecott," which for this report also includes Bear Creek Mining Company, Kennecott's U.S. exploration subsidiary), conducted a ground-magnetometer survey over the district, which revealed a strong magnetic high over the present property area. The survey was conducted with a portable Schmidt-type vertical-intensity magnetometer as part of a program of vehicle magnetics along county roads in Utah. Kennecott conducted grid magnetics and geologic mapping in the summer of 1954. In March 1955, Jensen recommended that Kennecott acquire the property; in March 1957, Frank Howd staked 35 claims in his name but on behalf of Kennecott. Between 1953 and 1957, Kennecott completed ground gravimetric, induced polarization ("IP"), and magnetic surveys as well as geochemical sampling and geologic mapping on the property. Kennecott dropped the claims in 1958. Kennecott's holdings did not include the Utah mine, which was owned by the LDS church through the Utah Mine Company, who had leased it to a John Fritch in 1953 in a 20-year lease-purchase option.

Upon learning that Kennecott was going to drop their claims and with Kennecott's permission, Jensen and M. P. Erickson re-staked the ground as the Crypto claims in the summer of 1958 on behalf of Pinnacle Exploration Inc. ("Pinnacle"), a subsidiary of Callahan Mining Corp. Robert Keeny, on behalf of former Kennecott employee Spenster M. Hansen, also located Golden Boy claims on the same ground at about the same time. At some point in 1958 or 1959, Pinnacle also completed a lease-purchase agreement with Fritch, the leaseholder of the Utah mine property. Pinnacle added new claims to the property in 1958 and 1959. Kennecott quitclaimed all rights to all Crypto claims to Jensen and Pinnacle in August 1958, apparently to circumvent the problem with the potentially conflicting claims of Spenster Hansen, according to the Utah International summary.

In July and August of 1958, Pinnacle conducted heavy-metal geochemical sampling, geological mapping, and a VLF-EM geophysical survey on the Crypto property. The heavy-metal geochemical sampling was not considered helpful, but the geophysical survey did identify several clustered EM conductors. According to the Utah International summary, Hansen probably drilled two holes, perhaps rotary holes, also in the summer of 1958, but there is no record of where they were located or the results of the drilling. In the summer of 1959, Pinnacle drilled two core holes totaling 228.6m (C-1, C-2). According to the Utah International summary, Pinnacle's holes eliminated an EM anomaly; although no magnetite was encountered, some iron oxide zones assayed weak Pb, Cu, and Ag.

In August 1961, Utah Construction & Mining ("Utah"), a predecessor company to Utah International Inc. (also referenced further as "Utah"), leased the Crypto claims and the Utah mine lease from Pinnacle, after carrying out a ground-magnetic survey with an Askania magnetometer to test the property's potential to host an economic magnetite deposit. There was no prior connection between Utah International and the Utah mine located on the property. In 1965, after Pinnacle defaulted on its agreement with John Fritch, Utah made a new and separate lease-purchase option agreement with Fritch for the Utah mine property. In 1967, Pinnacle's interest in the Crypto claims and the Utah mine lease was quitclaimed to Utah. In 1973 following considerable legal argument, the Utah Mine Co. conveyed ownership of the Utah mine to Fritch, who then immediately transferred it to Utah. Between 1961 and 1985, Utah completed various geological, geochemical, and geophysical surveys, including detailed ground magnetics and IP-resistivity. In addition, Utah drilled 39 core holes totaling 16,555.8m and eight reverse circulation ("RC") holes totaling 609.5m. The Main Zone sulfide zinc and oxide deposits were discovered during this time. In addition to their surface work, Utah also carried out underground mapping and sampling on two levels of the historic Utah mine (Shaw and O'Toole, 1975; Shaw, 1979; Hehn, 1979-1983).

Noble Peak Resources Ltd. ("Noble Peak") purchased the property from Utah in 1985 and compiled the existing drill-hole data, carried out a small soil and rock geochemical survey, and sampled the old drill core and mine dumps for their potential to support a silver-leaching operation.

In 1990, a joint venture between Cyprus Minerals Company ("Cyprus") and Mitsui Mining & Smelting Co. Ltd. ("Mitsui") obtained an option to earn a 50% interest in the property from Noble Peak. In the second half of 1990, Cyprus completed 15.3 line-km of gradient-array IP-resistivity and 3.2 line-km of dipole-dipole IP surveying along with surface geological mapping. The gradient-array IP-resistivity survey was conducted by Great Basin Geophysical with a line spacing of 122m and a dipole spacing of 61m. It located the main Crypto anomaly, its continuation to the east toward and under the Galena and Utah mines, and a new doughnut-shaped anomaly in the northeastern quadrant of the survey area (Cyprus/Mitsui Joint Venture Geologists, 1990). The dipole-dipole IP survey confirmed the presence of most of the major IP anomalies from the gradient survey. Cyprus re-logged 7,620m of Utah's diamond drill core and constructed detailed geological cross-sections. By the end of 1991, Cyprus had completed 17 diamond drill holes totaling 9,434.6m and two RC holes totaling 670.6m (Cyprus/Mitsui Joint Venture Geologists, 1990; Bernardi and Ohlin, 1991a, 1991b). Two of the diamond core holes (CCC-6B and CCC-10A) were wedges off existing holes that were discontinued due to significant down hole deviation. Among other things, this drilling confirmed the presence of and expanded the Deep Zone. Mitsui left the joint venture in 1991. Also in 1991, after completing

14 of the diamond drill holes (CCC-1 to CCC-12, including the two wedge holes) and the two RC holes (RCCC-1 and RCCC-2), Cyprus completed a "pre-feasibility study" based on a preliminary resource estimate and some bench-scale metallurgical test work on both the oxide and sulfide mineralization; their metallurgical testing is described in Section 16. They subsequently drilled an additional three core holes (CCC-13 to CCC-15) into the Deep Zone and extended known mineralization. Cyprus dropped their option on the property from Noble Peak in 1993.

In 1994, Noble Peak carried out a small prospecting and surface-rock-geochemical program to investigate the possibility of zone(s) of gold enrichment. According to Lithic, at some point between 1993 and 1996 the original unpatented Crypto claims were allowed to lapse; in March 1996, North American Exploration staked 54 Crypto Zn claims on behalf of Noble Peak, 40 of which have remained active and are part of the current property. In 1998, Noble Peak changed its name to Vaaldiam Resources Ltd. ("Vaaldiam"), began to concentrate on diamond exploration, and optioned the property to Sierra Gigantes Resources Inc. ("Sierra"). Sierra carried out an "enzyme leach" soil-sampling survey prior to dropping their option due to financing difficulties.

EuroZinc Mining Corporation ("EuroZinc") purchased the Crypto property from Vaaldiam in 2001 by purchasing a 100% equity interest in N.P.R. (US), Inc., a Nevada corporation and wholly owned subsidiary of Vaaldiam whose sole asset was the mineral title to the Crypto property. Other than compiling some of the historic results in a computer database, EuroZinc did not conduct any work during their tenure.

Lithic purchased N.P.R. (US), Inc. from EuroZinc in 2005, thereby acquiring the Crypto property. At the time of Lithic's purchase, the property included 40 unpatented mining claims (the Crypto Zn claims in Appendix A), partial or complete interest in 17 patented claims, and the state mineral lease, all of which were held in the name of NPR. Since then, Lithic has staked an additional 119 claims (the Crypto and Pony claims), purchased additional interests in two of the patented claims, and purchased a partial interest in one additional patented claim. Lithic's exploration is described in Section 10.0, and the list of claims and the lease that comprise the property is in Appendix A.

6.2 Historic Mineral Resource Estimates

A number of historic resource estimates for the Crypto project have been made by various parties over the years and are summarized below. Where tonnages were originally reported in Imperial units, they have been converted to metric based on conversion factors listed in Section 2.2. The following information on these historic estimates is presented for historic information only and in the interest of full disclosure. The reader is cautioned that these historic resource estimates were made prior to the implementation of NI 43-101 reporting requirements, do not conform to those requirements, and should not be relied on as being indicative of a resource or a reserve with demonstrated economic viability. Where terms later defined in NI 43-101 were used in the historic record, such terms have been enclosed in quotation marks.

6.2.1 1976 Estimate by Utah International Inc.

In 1976, Utah used a sectional approach to estimate what they termed a "proved resource" (Gorman and Jones, 1981). All of the mineralization included would be located in what is now termed the Main Zone between depths of 15 and 500m below surface. Fifteen separate zones involving four types of mineralization were outlined based on a 3.05m mining width and a minimum recovered metal value (Zn, Ag, Cu) of US$35/ton. Metal prices were current as of 1975, and estimated mill-recovery factors were 95% for sphalerite, 70% for sulfide-oxide material, and 50% for other minerals. Mill-recovery cutoff was 1% for Zn. Zinc mineralization was considered "oxide ore" if the ZnO/Zn ratio was greater than 0.1. "Protore" was defined as material comprising the "transition between barren and sulfide/oxide ore." Zones were geologically constrained by dikes and faults. Table 6.1 summarizes their estimate.

Table 6.1 Summary of 1976 Historic Resource Estimate by Utah International Inc.
(From Shaw, 1976)

Zone	Tonnes	% Zn	% Cu	gpt Ag	% Cd	% Pb	% Fe
oxide	1,492,000	9.11	0.21	33.6	0.094	0.210	24.00
sulfide	3,067,000	7.87	0.16	4.1	0.070	--	20.33
copper	103,000	2.26	3.44	35.3	0.016	--	8.95
"protore"	1,037,000	5.14	0.10	15.4	0.040	--	24.17
Total w/o protore	4,661,000	8.14	0.25	14.4	0.076	0.067	21.25
Total w/ protore	5,698,000	7.60	0.22	14.4	0.070	0.055	21.79

Note: Tonnage and silver grade have been converted to metric units; tonnes rounded to nearest thousand.

6.2.2 1991 and 1993 Historic "Geological Reserve" Estimate by Cyprus Minerals Company

Cyprus made a number of estimates of "geologic reserves" during their tenure on the property. In late 1990 after drilling the first three of their core holes, Cyprus estimated that the "geologic underground zinc sulfide reserves" at Crypto were 3.75 million tonnes grading 7.87% Zn (at a 2% Zn cutoff), 0.234% Pb, and 0.117 oz Ag/ton (Cyprus/Mitsui Joint Venture Geologists, 1990). This estimate was based on review of all available assay and geologic sections as well as plan-level maps that had been generated by prior operators combined with the results of their own early drilling. Open-pit oxide "reserves," which were based solely on the work of prior operators, were estimated to be 2.8 million tonnes averaging 7% Zn (Cyprus/Mitsui Joint Venture Geologists, 1990). Cyprus continued to update the "reserves" as they completed successive phases of drilling (Bernardi and Ohlin, 1991a, 1991b).

Cyprus estimated a "geological reserve" at Crypto using a cross-sectional method; details are given in Cyprus/Mitsui Joint Venture Geologists (1990). Sections were spaced at distances of 61.0 to 91.4m, and blocks were extended along section halfway between drill intercepts or to a maximum of 91.4m up or down dip in cases where no other drill intercepts were present. Blocks were then extended between sections to either half the distance to the adjacent section or to a maximum of 45.7m at the limit of drilling. Blocks were also constrained by geological features such as faults and dikes. Criteria used in the estimation were a cutoff grade of 2% Zn and a density factor of 9 cubic ft/ton (3.56 g/cm^3). No other ore elements were included.

Following the first part of their Phase III drilling in 1991, Cyprus re-calculated the "underground sulfide zinc reserves" for Crypto (Bernardi and Ohlin, 1991b). This 1991 "geological reserve," which was described in summary and commented on by Roscoe Postle and Associates in early 1993 (Agnerian, 1993), incorporated mineralization in both the Main and Deep Zones based on 25 core holes that intersected mineralized zones but did not include significant zinc mineralization encountered in three of the last four core holes drilled later in 1991 by Cyprus to test extensions of the Deep Zone. Drill-hole spacing of the 25 holes used was 91 to 152m (Agnerian, 1993). Subsequently in 1993, it appears that Cyprus re-estimated the resource and included the results of the three additional holes (Rockingham, 2001). Both estimates are summarized in Table 6.2.

Table 6.2 Summary of 1991 and 1993 "Geological Reserve" Estimates by Cyprus Minerals Company
(Modified from Cyprus/Mitsui Joint Venture Geologists, 1990;
Bernardi and Ohlin, 1991b; Rockingham, 2001)

Year	Zone	Tonnes	% Zn	gpt Ag	% Pb
1991	oxide	2,803,000	7.00	na	na
	sulfide	4,901,000	8.52	8.74	0.214
	Total	7,704,000	7.97	na	na
1993	oxide	2,803,000	7.00	na	na
	sulfide	5,442,000	8.68	na	na
	Total	8,245,000	8.11	na	na

Note: Tonnage and silver grade have been converted to metric units; tonnes rounded to nearest thousand

6.2.3 1995 Historic "Geological Resource" Estimate by Noble Peak Resources Ltd.

In 1995, Noble Peak commissioned an independent estimate from B. Henderson, who used a sectional approach to calculate a "geological resource" (Henderson, 1995). Blocks were defined on section for every drilled interval grading better than 2% zinc, ultimately including 34 drill holes. Blocks were extended halfway to adjacent drill holes along section or, if no other holes were present within 91.4m, to a maximum of 45.7m. These blocks were categorized as "probable." "Possible" blocks were added to the sections along which no drill information was available but where from adjacent sections it appeared likely that a zone of mineralization would occur. The dimensions of the "possible" blocks were proportional to those of the adjacent "probable" block, and the grades were the same. The blocks were then extended laterally to halfway between sections except where a dike or fault was present, in which case the block was terminated. An effort was made to group blocks into zones and/or "lenses" of mineralization, resulting in 20 lenses in eight zones.

According to Tindale (1997), "Henderson's reserve calculations expanded on the Cyprus calculations to include isolated intercepts in the Utah and Cyprus holes which in most cases could not be directly correlateable to assumed zones of mineralization." He further noted that "…Henderson's calculations do highlight the great quantity of zinc mineralization present in the Crypto deposit…"

Cumulative tonnage and grade figures were calculated for all blocks assigned grades of over 2% Zn and then for all blocks assigned grades of over 4% Zn. The results of these calculations are summarized by "category" and also in terms of sulfide *vs.* oxide mineralization in Table 6.3.

Table 6.3 Summary of 1995 Historic "Resource" Estimate by Noble Peak Resource Ltd.
(From Henderson, 1995)

	Tonnes	**% Zn**	**% Cu**	**gpt Ag**	**gpt Au**
2% Zn cutoff					
Total	15,693,784.0	6.17	0.271	13.97	0.170
"probable"	11,791,055.3	6.12	0.286	13.66	0.140
"possible"	3,902,728.7	6.29	0.226	14.93	0.259
oxide	5,065,530.1	6.11	0.427	23.31	0.141
sulfide	10,628,253.8	6.19	0.197	9.52	0.184
4% Zn cutoff					
Total	12,460,030.4	7.02	0.271	14.46	0.167
"probable"	9,094,430.9	7.09	0.285	14.02	0.122
"possible"	3,365,599.6	6.82	0.239	15.66	0.288
oxide	4,093,436.4	6.90	0.433	25.52	0.110
sulfide	8,366,594.0	7.07	0.191	9.05	0.194

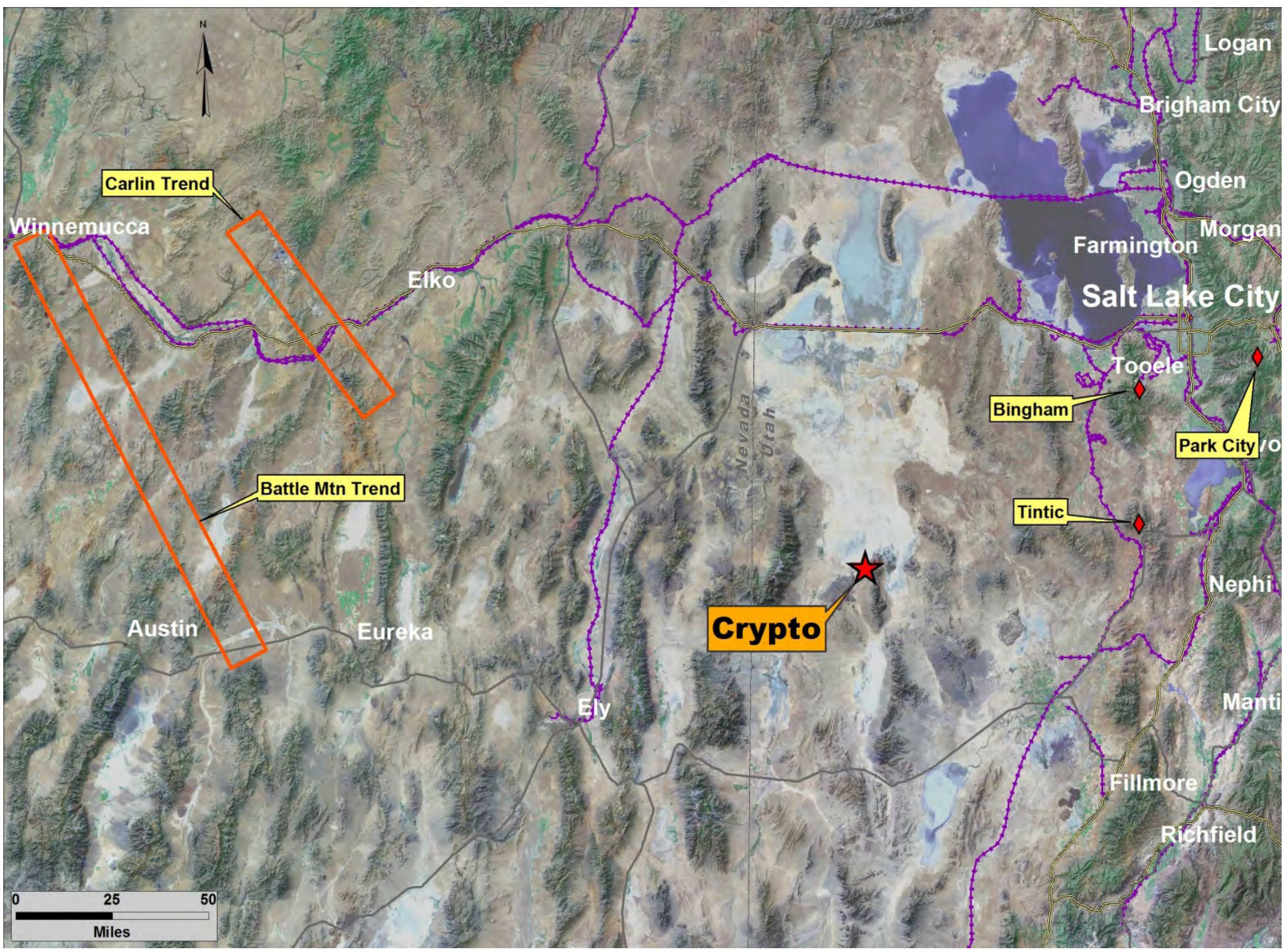
7.0 GEOLOGIC SETTING

7.1 Regional Geology

The following information on the regional geology is largely taken from Stokes (1986).

The Crypto zinc property is located on the northwestern edge of the Fish Springs Range, which is in turn located in the northeastern part of the Basin and Range Province of the southwestern United States (Figure 7.1). The Basin and Range physiographic province is characterized by generally north-trending fault-bounded mountain ranges and intervening basins that formed during regional Tertiary extension. The province is bounded on the east by Utah's Wasatch Range lying just east of Salt Lake City and on the west by the Sierra Nevada in eastern California.

Figure 7.1 Physiography of the Eastern Basin and Range Province



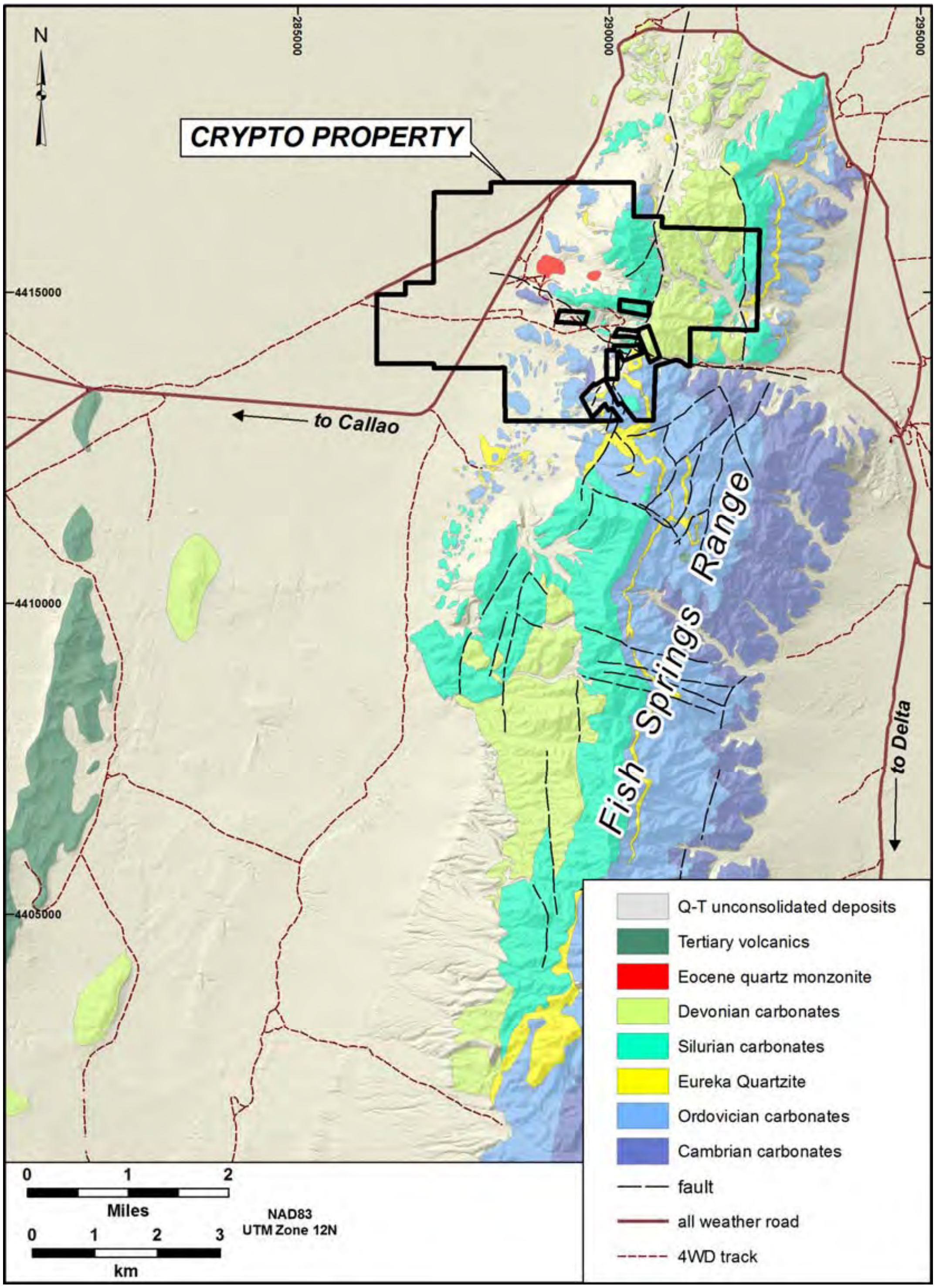
From Cambrian through Early Triassic time, western Utah was the site of marine deposition of shelf deposits in the Cordilleran geosyncline. East-directed Paleozoic compressional deformation that affected central and northeastern Nevada did not reach into west-central Utah. Mesozoic sedimentary rocks younger than the Early Triassic are only rarely exposed in western Utah, which appears to have been part of a poorly defined highlands area extending into Nevada at that time. The first intrusive igneous activity in western Utah since the Precambrian is marked by Jurassic intrusions with dates ranging from 140 to 152 Ma in the House Range, Silver Island Mountains, Gold Hill area, and Newfoundland Mountains that are unusual in being located in an extensive sedimentary basin. Late Cretaceous to early Tertiary folding, thrust faulting, and uplift of the Sevier and Laramide orogenies affected central and northeastern Utah but in western Utah have only limited exposures, e.g. Deep Creek Range near the Nevada border. Intermediate volcanic and intrusive activity was widespread in Utah during the middle Tertiary with exposures of these units in the ranges to the east and west of the Fish Springs Range. Intrusive rocks presumed to be Late Eocene in age found in the Fish Springs Range may be early examples of this igneous activity. A separate and distinct volcanic center was located in the approximate center of Juab County with exposures in Desert Mountain, Key Mountain, and the Thomas Range. Less extensive mafic volcanism also occurred during the late Cenozoic in western Utah.

7.2 Local Geology

Generally speaking, the Fish Springs Range is a north-trending horst comprised of Lower Cambrian to Upper Devonian platformal sedimentary rocks that have been homoclinally tilted to the west at generally moderate dips (Hintze, 1980; Lindsey *et al.*, 1989) (Figure 7.2). The sedimentary package, which exceeds 3,500m in total thickness, consists mainly of carbonate rocks, with minor interbedded shales and quartzites. Scattered quartz monzonite intrusive complexes and rhyolite dikes and occasional andesitic plugs intrude the sedimentary rocks locally and are presumed to be Late Eocene age. Tertiary volcanic rocks of latitic, shoshonitic, and rhyodacitic composition are found along with younger alluvium in basins adjoining the Fish Springs Range.

The general structure of the Fish Springs Range is relatively simple, the dominant element being a series of northerly trending extensional faults related to the development of the Basin and Range Province. Most are normal with west-side down and small to moderate displacements. Other than the Juab fault described below, faulting and/or folding are of minor significance within the northern end of the Fish Springs Range as evidenced by regional mapping to date.

Figure 7.2 Geology of the Fish Springs Range Showing the Crypto Property
(Provided by Lithic Resources Ltd., 2009)



7.3 Property Geology

The following description of the Crypto property geology is synthesized from various Utah (Hehn, 1979-1983; Shaw, 1979; Shaw and O'Toole, 1975) and Cyprus (Cyprus/Mitsui Joint Venture Geologists, 1990; Bernardi and Ohlin, 1991a, 1991b) reports, together with observations and interpretation by Lithic personnel and consultants. The property is predominately underlain by a sequence of lower Cambrian to upper Devonian platformal carbonate units, mainly dolostones with lesser, generally thin-bedded limestones and minor interbedded quartzites and shales (Figure 7.3 and Figure 7.4). From youngest to oldest, the stratified rocks include the following divisions:

Lower Devonian Sevy Dolomite *– light gray, fine-grained, thin- to medium-bedded, banded dolostone*

Middle Silurian Laketown Dolomite;

Thursday Member – interbedded dark to pinkish gray, medium to coarsely crystalline dolostone;

Lost Sheep Member – light olive gray, fine to medium crystalline dolostone in lower portion; medium dark gray dolostone with 10% chert as nodules and stringers;

Harrisite Member – thick-bedded, medium dark gray, finely crystalline dolostone with 10-15% dark brown chert as nodules and stringers;

Bell Hill Member – banded, light to dark gray, finely to coarsely crystalline dolostone

Upper Ordovician Ely Springs Dolomite;

Floride Member – light gray, thin to medium bedded, finely crystalline limestone;

Lower Member – also known as Fish Haven Dolomite – medium-bedded, medium gray, finely crystalline limestone in lower portion; dark gray, cliff-forming dolostone which may be attenuated in places in upper portion.

Middle Ordovician Eureka Quartzite *– thin- to thick-bedded, shattered, white to light gray, mediu- to fine-grained orthoquartzite and quartz sandstone;*

Watson Ranch Quartzite – yellowish-brown, friable, calcareous sandstone with fucoidal markings

Lower to Middle Ordovician Pogonip Group;

Deadman Springs Dolomite – thin- to medium-bedded, light reddish brown, sandy dolostone;

Kanosh Shale – olive-gray shale with interbedded fossiliferous, thin-bedded limestone;

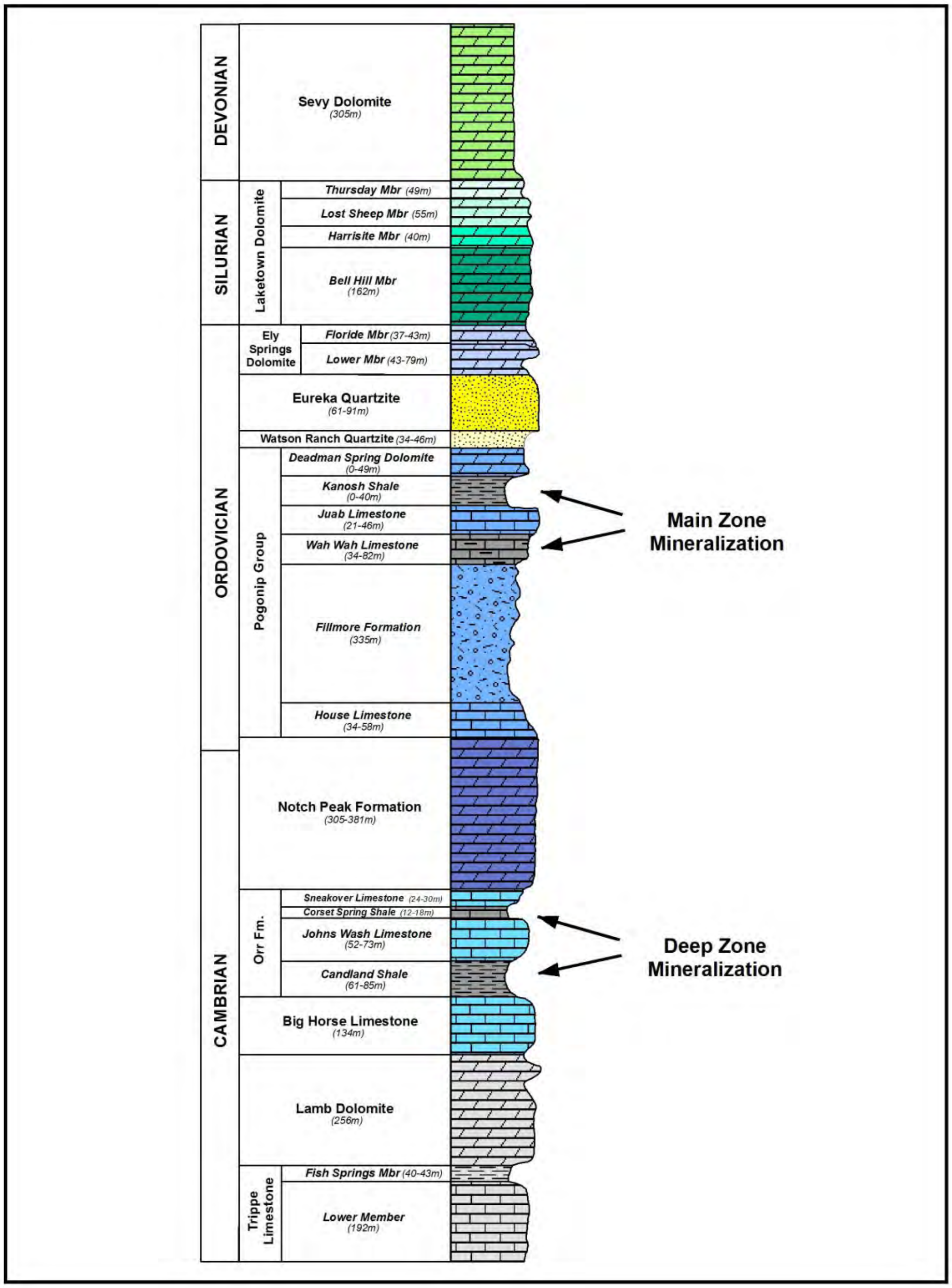
Juab Limestone – medium- to thick-bedded, medium gray, cliff-forming limestone;

Wah Wah Limestone – very thin- to medium-bedded, medium gray, silty limestone with interbedded, siltstone and intraformational conglomerate;

Fillmore Formation – thin-bedded, medium gray intraformational conglomerate interbedded with light olive-gray, silty shale;

House Limestone – thin- to medium-bedded, light bluish gray, finely crystalline limestone.

Figure 7.3 Stratigraphic Column for the Crypto Property
(Modified from Mitchell and Ohlin, 1991)



Lower Ordovician to Upper Cambrian Notch Peak Formation *– medium- to thick-bedded, dark gray, cliff-forming dolostone with some thin interbeds of medium to light gray limestone near top*

Upper Cambrian Orr Formation;

Sneakover Limestone Member – medium to light gray limestone in bottom portion; thin- to medium-bedded silty limestone in upper portion;

Corset Spring Shale Member – greenish shale with interbeds of thin-bedded dark grey limestone;

Johns Wash Limestone Member – medium to dark gray limestone; upper portion dolomitic;

Candland Shale Member – olive-gray shale interbedded with dark grey, fossiliferous, nodular limestone;

Big Horse Limestone – medium gray, partly oolitic in lower portion; thin-bedded in middle portion; dark gray, mottled, partly bioclastic in upper portion; medium-gray, thin- to medium-bedded near top.

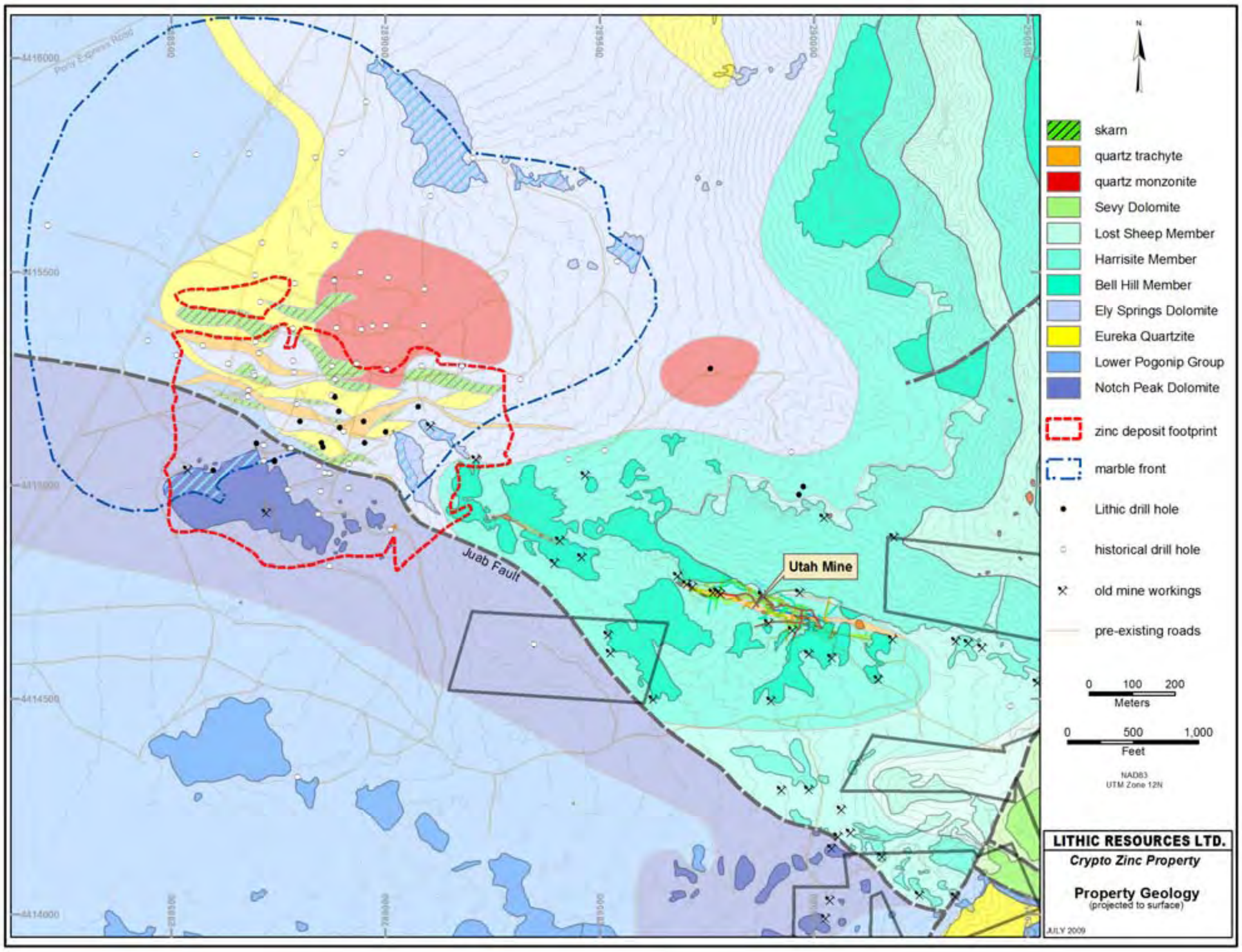
Outcrop mapping and observations by Lithic geologists coupled with mapping and core logging by Utah and Cyprus geologists indicate that where exposed at surface, the sedimentary rocks strike northerly and have a shallow to moderate westerly dip. In some places, minor northerly trending folds result in a degree of flattening or even dip reversals to the east.

A number of faults have been mapped or are postulated to exist on the property, the most obvious being the Juab fault, a west-northwest-trending, north-dipping normal fault that trends through the middle of the property. The fault is significant enough to have caused an apparent left-lateral offset of the Fish Springs Range and is thought to have on the order of 500-600m of net vertical displacement. The north-trending Overland fault along the western margin of the Fish Spring Range, together with a number of lesser subparallel faults, are normal with west-side down and represent extensional faulting typical of the Basin and Range Province. Various other lesser faults with varying orientations have been interpreted on the property in the past, but the evidence for them is not as clear.

Drilling has revealed that the sedimentary package has been intruded by a late Eocene (38.5 ±1.0 Ma, K/Ar) felsic intrusive complex which underlies a large part of the property at depth but is not exposed at surface. In the vicinity of the Crypto zinc deposit, it rises to the bedrock surface in a cupola which sub-crops below shallow Pleistocene gravels. Two main phases have been identified so far: a grey, medium-grained, equigranular to weakly porphyritic biotite quartz monzonite and a pinkish buff, medium-grained, equigranular biotite quartz syenite.

Various dikes ranging in composition from porphyritic quartz trachyte to rhyolite are exposed in outcrop or have been observed in drill core and are thought by Lithic geologists to be related to the same intrusive event as the quartz monzonite. The most obvious is the so-called Utah dike, which averages about 15m in width at surface and can be traced for over a kilometer easterly from the Crypto deposit through the area of the Utah mine. Other east-trending dikes of similar composition, some up to 30m wide, have been intersected in drilling.

Figure 7.4 Geologic Map of the Crypto Zinc Property
(Provided by Lithic Resources Ltd., 2009)



Thin-bedded carbonate rocks near the intrusive complex, particularly those with shaly partings, have been altered to skarn, marble, and siliceous hornfels. More thickly bedded to massive dolostones tend to be converted to marble. Although dips are shallow to moderate where observed in outcrop, drilling has shown that bedding attitudes in the immediate vicinity of the intrusion have been disrupted such that they can have very steep dips. This has probably resulted from stoping and doming by the intrusion, although drag on the Juab fault may also have been a factor. Thin-bedded limestones of the Orr Formation immediately south of the Juab fault have been folded at depth such that dips near the fault are close to vertical.

Zinc-copper-indium mineralization of the Crypto deposit occurs in portions of the skarn, while molybdenite mineralization occurs both as disseminations in the skarn and in more classic, porphyry-style quartz-pyrite veinlets within the quartz monzonite intrusion itself. High-grade lead-zinc-silver mineralization, including that exploited in the historic Utah and Emma mines, occurs in structurally controlled replacement zones in carbonate rocks peripheral to skarn.

The western half of the Crypto property is mostly overlain by Pleistocene lacustrine deposits from glacial Lake Bonneville, which show wave-cut terraces at a number of levels where they lap against the Fish Springs Range.

8.0 DEPOSIT TYPES

The principal mineral deposits thus far identified in the Fish Springs mining district formed as part of a large skarn/carbonate replacement system. In general, skarns are formed when hydrothermal fluids containing silica, various metals, and other dissolved components are introduced into carbonate host rocks, resulting in the formation of calc-silicate minerals such as garnet, diopside, epidote, biotite, chlorite, amphibole, wollastonite, and idocrase. In more magnesian hosts, humite, periclase, and olivine may be present. Iron oxides such as magnetite and sulfides such as chalcopyrite, sphalerite, pyrite, pyrrhotite, galena, and arsenopyrite are deposited in the system, along with variable amounts of gold and silver. Skarn deposits are often, but not always, associated with intrusive rocks and are generally, but not always, developed in carbonate rocks. They have been classified into seven types on the basis of metallic affinity: Fe, Au, Cu, Zn, W, Mo, and Sn (Meinart *et al.*, 2005).

As defined by Meinart *et al.*, zinc skarns are usually related to oxidized dioritic to high-silica rhyolitic intrusions that may be barren or contain sub-economic copper and/or molybdenum ± tungsten ± tin deposits. The skarns are mined primarily for zinc-lead-silver ores and commonly contain recoverable copper, gold, molybdenum, and tin and/or tungsten. Ore is found in the form of stratabound mantos or cross-cutting, chimney-style replacements in limestone and dolostone, often distal (up to several kilometers) from their genetically associated intrusions. Mineralogical and chemical zonations within the skarns are well developed, but contact-metamorphic aureoles are usually absent or weakly developed. In addition to their metal associations, zinc skarns are generally distinguished from other skarns by a distinctive suite of iron and manganese-rich alteration minerals, including olivine, chlorite, pyroxene, garnet, serpentine, amphibole, pyroxenoids, and ilvaite; by the lack of a significant metamorphic aureole centered on the skarn; and by their occurrence along structural and lithologic contacts.

Carbonate replacement deposits (“CRD”) have been described by Megaw (1998), Titley (1993), and others as a broader category of high-temperature, carbonate-hosted massive sulfide deposits within which zinc skarns could be placed, but which would also include systems with no calc-silicate development. Contacts between sulfide mineralization and host rocks are generally very sharp, and replacement textures are typical. They generally display a simple, polymetallic assemblage of galena, sphalerite, chalcopyrite, arsenopyrite, and pyrite or pyrrhotite along with carbonate, sulfate, fluorite, and quartz gangue. They also may contain recoverable molybdenum, tungsten, tin, cadmium, gallium, germanium, and indium. Examples are widespread throughout the Cordillera of North and South America, including the zinc-rich Gilman and Leadville deposits in Colorado, the Midway and Ketza River deposits in Canada, and the Charcas, San Martin, and Bismarck deposits in Mexico.

The principal mineral deposits thus far identified on the Crypto property are related to a felsic intrusion of Late Eocene age. A number of Utah’s most significant historic mining camps, from which large-scale mineral production has been derived, are based on carbonate replacement and skarn deposits related to similar felsic intrusive systems of late Eocene to early Oligocene age (Krahulec, 2007). Examples include the Main Tintic, Bingham, and Park City districts about 115 to 190km to the east of the Crypto property. In the case of Bingham, the replacement deposits are peripheral and related to the world-class Bingham porphyry copper deposit (Table 8.1). Weaker porphyry systems are also associated with the intrusive bodies at Tintic and Park City.

Table 8.1 Selected Carbonate Replacement Deposit Production in the Western USA
(From Titley, 1993)

District	Production (t)	% Cu	% Pb	% Zn	gpt Ag	gpt Au
Bingham, UT – carb. replacement only	13,476,233	0.44	8.8	3.4	167	2.35
Bingham, UT – all non-porphyry incl. skarn	39,868,000	0.93	4.7	1.9	106	1.85
Tintic, UT	17,521,000	0.90	5.9	1.2	485	4.86
Park City, UT	13,300,000	0.38	8.7	4.5	556	2.28
Gilman, CO	10,586,849	0.90	1.5	8.5	228	1.70

9.0 MINERALIZATION

The most significant mineralization discovered to date on the Crypto property consists of sphalerite with minor chalcopyrite occurring in a series of concordant to discordant skarns and replacement bodies in carbonate rocks south of and adjacent to the quartz monzonite intrusive complex. Two main types of skarn have been distinguished on the basis of mineralogy, generally reflecting the chemistry of the host rock. The most common type is magnesian, consisting of humite ± magnesioferrite ± phlogopite along with lesser spinel, periclase, actinolite, forsterite, and tremolite. Humite and forsterite may be partly retrograded to serpentine, brucite, and/or talc. Phlogopite may be partly altered to chlorite, while periclase may be converted to brucite. Magnesioferrite, a Mg-rich member of the magnetite group of minerals, is very abundant and often massive. Magnetite has also been identified, but a significant proportion of what has been historically classified as magnetite in the Crypto area is probably magnesioferrite.

A second and less common type of skarn is more calcareous in composition. It generally exhibits a less disrupted character, with preserved bedding replaced by alternating bands of reddish brown grossularite garnet separated by bands of fine-grained diopside and potassium feldspar, probably reflecting a protolith of thinly bedded limestone with shaly partings. Magnetite/magnesioferrite is occasionally present.

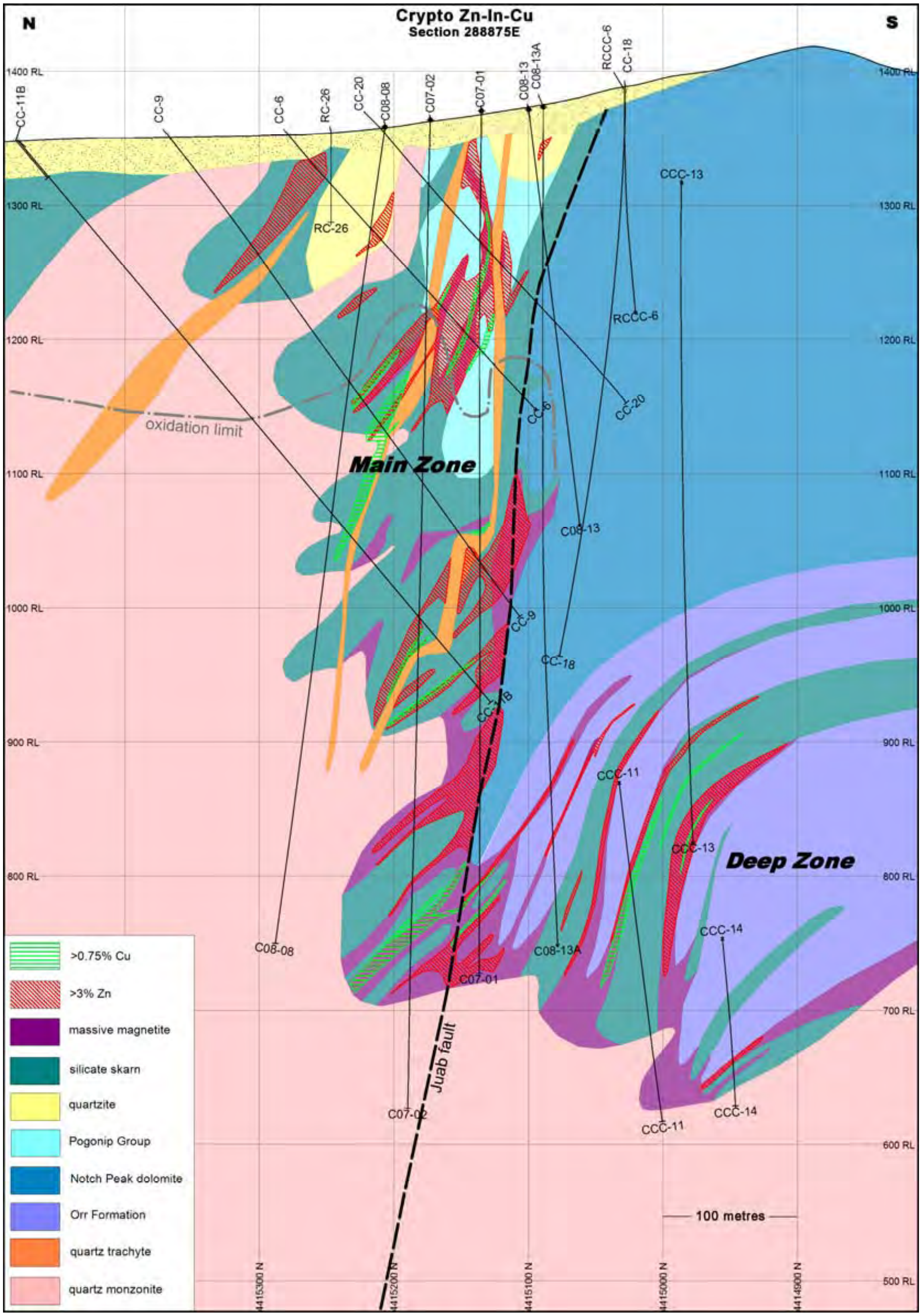
Two main areas of zinc mineralization have been found, the Main and Deep zones, neither of which is exposed in outcrop. The two zones are separated by the Juab fault. Figure 9.1 illustrates a typical cross-section in the Crypto deposit area.

9.1 Main Zone

The Main Zone occurs in Ordovician Pogonip Group carbonate rocks, and possibly in some Ely Springs Dolomite, situated within the hanging wall north of the Juab fault and immediately south of the sub-cropping quartz monzonite cupola. The Main Zone is bounded on the south by the Juab fault and the weakly mineralized and skarn-altered footwall Notch Peak Formation. In the western part of the deposit, sphalerite-chalcopyrite mineralization occurs in irregular bodies of humite-phlogopite skarn, with or without magnesioferrite. Magnesioferrite abundance declines in the eastern part of the zone, and the proportion of coarse-grained phlogopite increases as does the amount of pyrite. However, the overall abundance of pyrite and other iron sulfides such as pyrrhotite in the Crypto system is quite low.

Sphalerite in the Main Zone is medium to coarse grained and red-brown to brown in color. Petrographic work shows that it typically occurs as relatively coarse aggregates up to about 1cm across composed of roughly equant anhedra with a mosaic texture. Individual grains are generally 0.1 to 1mm in size but can range up to 5mm in diameter. Sphalerite also occurs intermixed with or interstitial to magnesioferrite. Sphalerite often contains small amounts of chalcopyrite and pyrrhotite as tiny grains along crystal grain boundaries, randomly or crystallographically oriented arrays of grains within a single sphalerite grain, or as irregular anhedra. Chalcopyrite inclusions in sphalerite may be partly altered to bornite or, rarely, chalcocite. Very small amounts of pyrite and pyrrhotite are found scattered throughout zones of sphalerite mineralization.

Figure 9.1 Cross-section of the Crypto Deposit: Looking East
(Provided by Lithic Resources Ltd., 2009)



Where it does not occur within sphalerite, chalcopyrite tends to be found as disseminated grains and blebs ranging from a few millimeters to a few centimeters in diameter. Copper grades are not directly proportional to those of zinc, its distribution typically only partially overlapping with that of zinc in any given drilled interval. However, there appears to be at least a rough increase in the Cu/Zn ratio with proximity to the intrusion, and discrete zones of copper enrichment are definable.

To date, Main Zone mineralization has been traced with drilling over a length of about 525m, a width of about 150m, and to a depth of 575m and remains open to the west and to depth. Because of strong alteration, probable disruption of bedding by intrusive stoping, and a lack of suitable marker horizons, stratigraphic correlations and the degree of discordancy of skarn and replacement mineralization in this area are difficult to ascertain. In addition, original sedimentary features have been largely obliterated within skarn zones, which may include chaotically disrupted and brecciated textures. However, it appears that bedding tends to dip steeply to the north in the immediate vicinity of the deposit and that mineralization is preferentially developed in more thinly bedded units, probably the Kanosh Shale and Wah Wah Limestone, although portions of the Ely Springs Dolomite may also be involved. More thickly bedded or massive units in the section, particularly dolomitic ones, are more likely to have been marbleized.

The upper part of the Crypto system includes a number of extremely siliceous zones, at least some of which are probably tilted (stoped) blocks of Eureka Quartzite. However, the overall volume of siliceous rock appears to be greater than would be suggested by nearby undisturbed occurrences of this unit, and thus some parts of these zones may represent massive, pervasive silicification of dolostones and limestones resulting in a fractured, dark grey, granular, highly siliceous rock. Quartz-rich zones in the upper portions of the intrusive cupola may, in part, represent partly consumed and dismembered rafts of Eureka Quartzite.

The Main Zone has been oxidized to an average depth of about 250m. Sphalerite and chalcopyrite have been converted to a mixture of smithsonite, hemimorphite, hydrozincite, and zincite, with lesser wulfenite, covellite, franklinite, malachite, and gageite in a matrix of various iron and manganese oxides. The deposit is overlain by Pleistocene gravels ranging in thickness from 2-40m.

9.2 Deep Zone

The Deep Zone is located immediately south of the Juab fault and is hosted predominantly within thinly bedded limestones and shaly members of the Orr Formation, probably the Corset Spring Shale and Candland Shale members. Stratigraphic correlations are much clearer here due to less apparent structural complications within the Juab fault footwall. Mineralization also occurs as proximal skarn along the contact of the Orr Formation and the underlying quartz monzonite intrusive. Mineralization consists of coarse-grained, reddish sphalerite with minor disseminated chalcopyrite, pyrite, and/or pyrrhotite that are found in stratabound bodies of semi-massive to massive magnesioferrite/magnetite intercalated with humite ± periclase skarn, grossularite-diopside-K feldspar skarn, and marble. The marble tends to contain abundant fracture-fillings of magnesioferrite in proximity to massive magnesioferrite layers. Humite within magnesioferrite-rich zones tends to be serpentinized. Significant portions of the massive to thick-bedded dolostones of the overlying Notch Peak Formation have been converted to marble.

At least three separate mineralized horizons have been identified through drilling over an area of about 330m by 225m at depths of between about 450 to 750m. They remain open to the west, south, and possibly the east.

9.3 Skarn-Hosted Indium and Molybdenum

Indium is present in significant quantities in the Crypto deposit. Initial petrographic work and energy dispersive X-ray ("EDX") analysis have shown that it occurs in the sphalerite lattice in amounts as high as an exceptional 9% by weight along with minor amounts of cadmium and manganese. Sphalerite with high indium content tends to be green in color and typically shows "chalcopyrite disease" textures; Figure 9.2 shows an extreme example of high-indium sphalerite. Despite its essentially exclusive occurrence in sphalerite, there is not a direct relationship between zinc grade and indium grade. The highest indium grades tend to occur in mineralization with a zinc grade of between 0.5 and 5%.

Numerous intervals of skarn mineralization have been found to contain significant levels of molybdenum, although it is not obvious macroscopically. Petrographic work has shown that it occurs as small scattered laths associated with magnesioferrite grains in partly serpentinized humite, apparently as a primary constituent of skarn.

Finally, small amounts of two bismuth minerals tentatively identified as bursaite ($Pb_5Bi_4S_{11}$) and tsumoite (BiTe) have been identified in skarn from the eastern part of the deposit. Analytical data show that in addition to the main elements of interest – Zn, Cu, In, and Mo – trace elements enriched to varying degrees in skarn mineralization include Cd, Ag, Mn, Sn, Co, and Au.

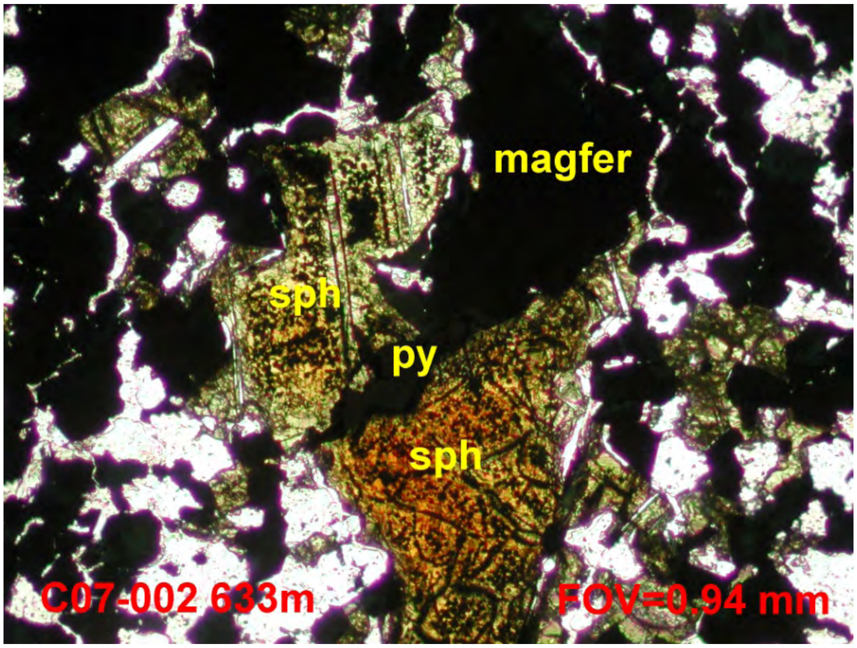
9.4 Other Skarn Occurrences

Utah drill hole CC-43 intersected a 3.05m interval grading 7.65% Zn, 3.50% Cu, and 0.100% Mo within a 7.6m interval of semi-massive to massive sulfides at a depth of 889m and a horizontal distance of about 650m to the east of the Main Zone. Pyrite, chalcopyrite, sphalerite, and molybdenite in this interval occur as veins, masses, and strong disseminations in a thinly bedded limestone with shaly partings. A highly siliceous rock immediately above this interval was logged as quartzite, but its biotite and sulfide content as well as its location within an argillically and propylitically altered granite suggest that it may be a highly siliceous intrusive phase. The presence of disseminated magnetite over a 70m interval in the shaly limestone below the sulfide mineralization suggests incipient skarn development, potentially related to the main Crypto hydrothermal system.

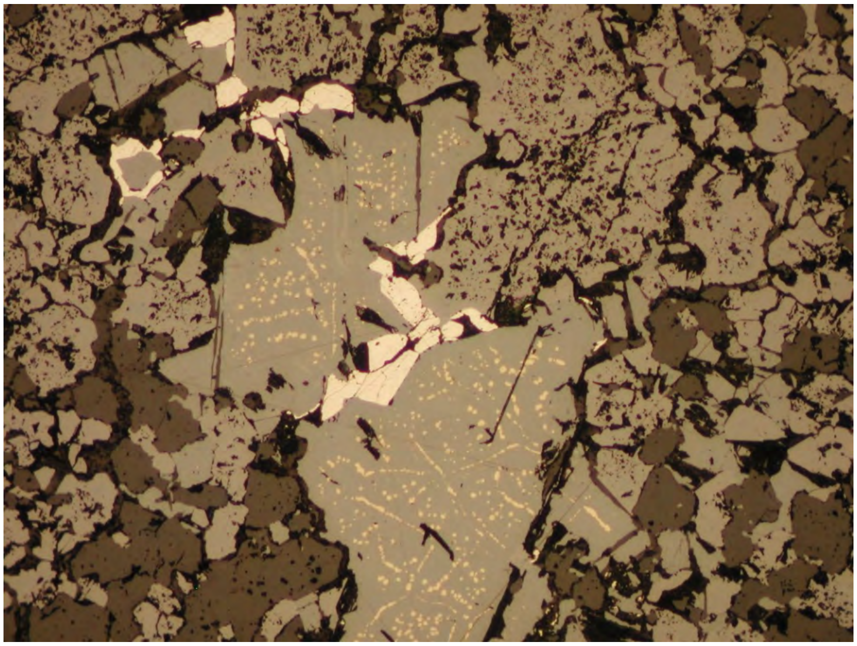
Although drilling north of the main intrusive cupola is sparse and generally shallow, several historic drill holes intersected narrow intervals of skarn zinc mineralization in this area, suggesting potential to discover additional resources here.

Figure 9.2 Sphalerite with High Indium Content Exhibiting "Chalcopyrite Disease" Textures

High-indium sphalerite from Main Zone – plane polarized light



High-indium sphalerite (same view as 9.2a) – reflected light



9.5 Replacement Style Ag-Zn-Pb

In the eastern portion of the property, silver-rich galena with minor sphalerite and pyrite occur in steeply dipping, structurally controlled, and somewhat discontinuous replacement zones within the Ely Springs and Laketown dolomites. The most significant zone outlined to date is that exploited by the historic Utah and Galena mines, where mineralization occurs in the vicinity of but not immediately adjacent to the Utah dike. A total of about 3,960m of workings on six levels were accessed via the 246.9m Utah shaft. The Galena mine immediately to the west was accessed via the 213.4m Galena shaft and included a total of about 915m of workings, one of which connected with the Utah mine on the 512ft (156m) level. Mine development extended between 152 and 243m to the west and east respectively of the Utah shaft. Minor production of similar mineralization was derived from the Emma mine and to a much lesser extent from a number of small excavations to the southeast, including the Wilson and Carnation mines. Most of the total production from the Fish Springs district between 1890 and 1953 came from the replacement mineralization in the Utah, Galena, and Emma mines (Perry and McCarthy, 1976).

Underground mapping by Utah on the uppermost 269- and 440ft (82- and 134m) levels of the Utah mine indicated that the best mineralization occurred in the central part of the mine where a pipe-like structure with a diameter of approximately 60m resulted from the intersection of fault structures. Mineralization consisted of irregular pods of anglesite, cerussite, smithsonite, covellite, willemite, malachite, hemimorphite, aurichalcite, and wulfenite in a matrix of iron and manganese oxides situated adjacent to, but generally not within structures. The cores of larger masses of ore as well as mineralization below the water table in the deepest parts of the mine showed that these oxidation products were derived from a relatively simple sulfide mineralogy of galena and pyrite with minor sphalerite and chalcopyrite.

Gorman and Jones (1981) noted that silver and lead grades tended to increase with depth and reported that grades above the 440-level (134m level) averaged 5.5% lead and 754g Ag/t, while the grades on the lowermost 812-level (247m level) were 24% Pb and 1,166g Ag/t. A deep drill hole (CC-43) completed by Utah below the Utah workings intersected a 2.87m interval grading 6.8% Zn, 3.8% Pb, and 167.1g Ag/t approximately 230m below the lowermost mine level, indicating that mineralization continues to depth.

Lithic drill hole C08-07A was aimed to test the down-plunge extension of the Utah mine zone between the lowermost mine workings and the intercept in hole CC-43. It unexpectedly intersected unmineralized Eureka Quartzite at the point where mineralization was projected to be present. The Eureka Quartzite is not mineralized elsewhere on the property and is not considered to be a favorable host for the development of skarn or replacement-style mineralization. It is not a thick unit and as a result, the potential for additional Utah zone mineralization in the adjacent carbonate rocks remains.

Lithic drill hole C08-11 intersected a wide interval of replacement-style silver-zinc-lead mineralization to the east of the Main Zone, including a 10.83m interval grading 253.8g Ag/t, 4.28% Zn, and 0.68% Pb. Numerous other intercepts of high-grade silver mineralization within the oxide zone were encountered in historic drilling, including 10.21m grading 207.8g Ag/t, 12.81% Zn, and 0.98% Pb in hole CC-34 and 3.56m grading 361.5g Ag/t, 10.19% Zn, and 12.95% Pb in hole CC-27. These intercepts and others suggest significant potential for other high-grade replacement zones in the Crypto vicinity, similar to but distinct from that mined historically at the Utah mine.

9.6 Molybdenum

In addition to disseminations in skarn, molybdenite occurs in porphyry-style quartz-pyrite veinlets; larger, banded, quartz-pyrite veins; and as fracture coatings in the quartz monzonite intrusion underlying the Deep Zone. More intensely mineralized zones in the intrusion tend to be argillically altered and sericitized and include small amounts of oxidized, disseminated pyrite. The general distribution of mineralization on the Crypto property, with a lower molybdenum-bearing zone grading upwards and outwards through copper, zinc, and then lead and silver-rich zones with increasing distance from the quartz monzonite intrusion, suggests that mineralization at Crypto may be related to a large underlying porphyry molybdenum system. Agnerian (1993) reported that hole CCC-15 contained an intersection of 0.5% to 3% MoS_2 over 18.6m.

10.0 EXPLORATION

Lithic acquired the Crypto zinc property from EuroZinc in 2005. In continuing the compilation of historic results, Lithic found that all historic drill core had been lost.

Since 2006, Lithic has conducted exploration that included photogrammetry, a helicopter-borne magnetic survey, a pole-dipole IP survey, 10,639m of core drilling, and preliminary metallurgical testwork. In addition, the existing computer database was enhanced and corrected using original data records.

Lithic's drilling activities are described in Section 11.0; its metallurgical investigations are described in Section 16.0.

10.1 Photogrammetry

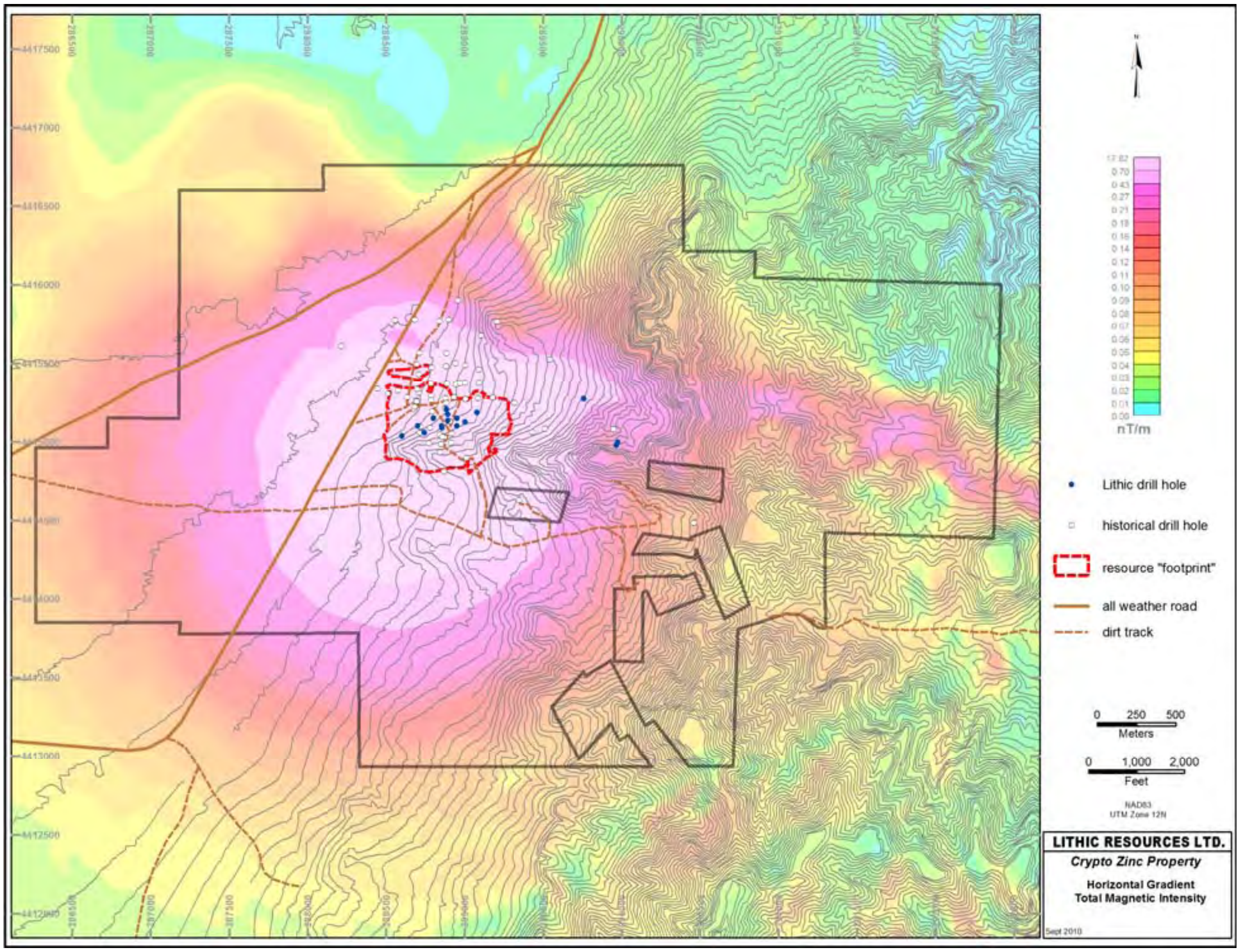
In 2006, Lithic contracted with Eagle Mapping to carry out a program of 1:8,000-scale color aerial photography over the general property area and to construct a detailed (2m contour) 1:2,000-scale topographic map and orthophoto of the central portion of the property. These maps were used as a base for further work.

10.2 Magnetic Survey

In 2006, McPhar Geosurveys Ltd. was contracted to carry out a high-resolution, helicopter-borne magnetic survey of the general property area. Approximately 1,018 line-km of survey were flown at a line spacing of 100m and a mean terrain clearance of 30m for the magnetometer. Deliverables included plots of total field magnetic, reduction to pole, first and second vertical derivatives, horizontal gradient, and analytic signal data.

Figure 10.1 illustrates the calculated horizontal gradient of total magnetic intensity in the property area and shows a massive positive magnetic anomaly centered on the Crypto deposit.

Figure 10.1 Calculated Horizontal Gradient of Total Magnetic Intensity – Crypto Area
(Provided by Lithic Resources Ltd., 2009)

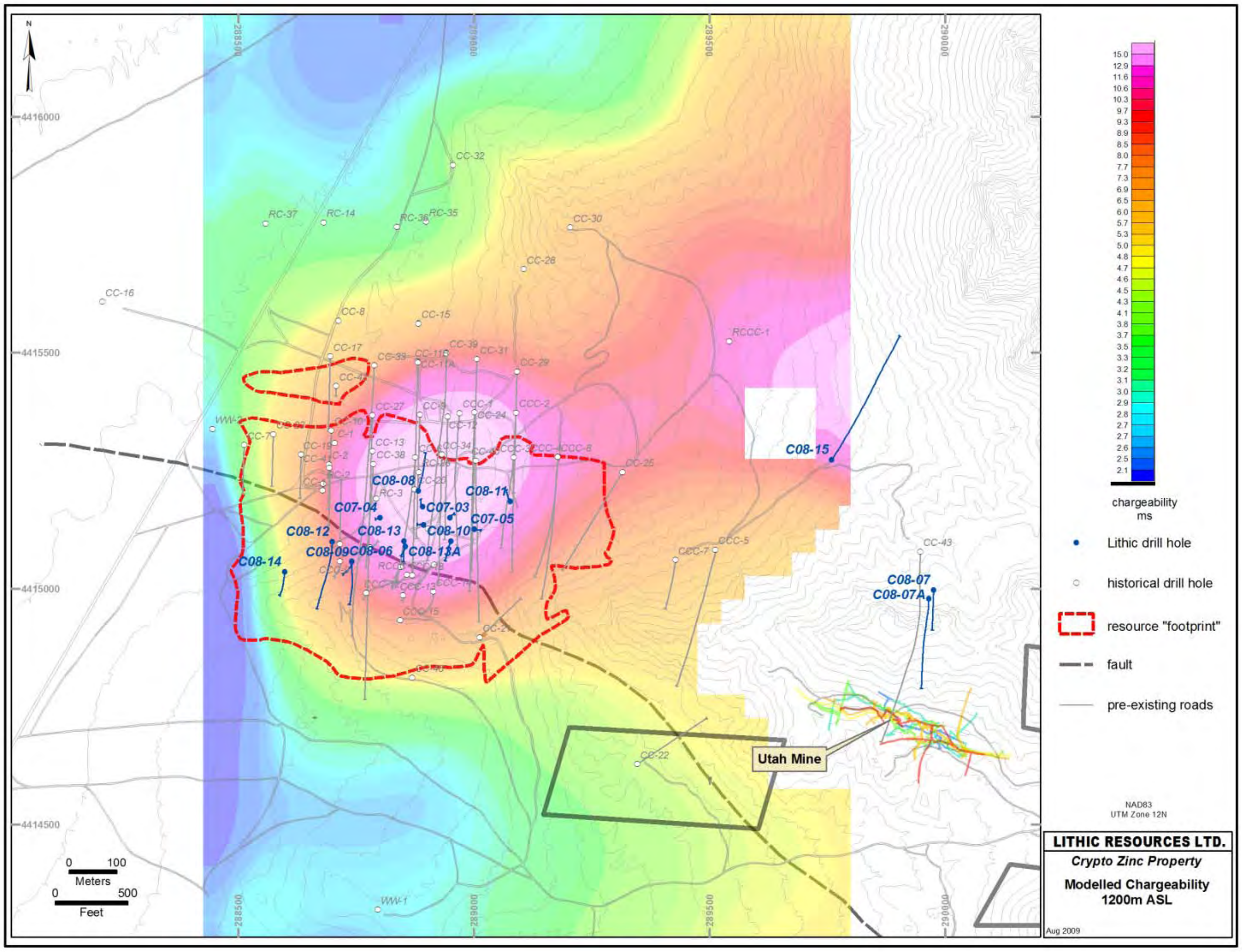


10.3 Induced Polarization Survey

In 2006, Peter E. Walcott and Associates Limited carried out a pole-dipole IP survey over the central part of the property. Approximately 25km of surveying on lines spaced 200m apart were completed using an "A" spacing of 100m. Deliverables included a series of pseudosections and plan views, including modeled data at elevations of 1200, 1300, and 1400m ASL.

The results of the survey showed a distinct chargeability anomaly apparently centered on known zinc mineralization, as well as a separate and similar but somewhat deeper anomaly located approximately 1km to the east. Figure 10.2 illustrates a plan of modeled chargeability values at an elevation of 1200m ASL. Lithic drillhole C08-15 targeted the eastern anomaly and intersected both intrusive rocks and sections of carbonates with varying development of skarn along with geochemically elevated levels of zinc.

Figure 10.2 Plan of Modeled Chargeability in the Central Part of the Crypto Property
(Provided by Lithic Resources Ltd.; Elevation is 1200m ASL)



11.0 DRILLING

11.1 Summary

Four campaigns of drilling involving a total of 85 core and RC exploration holes totalling 38,138m have been carried out on the Crypto property. Table 11.1 summarizes the drill-hole data in the database, while Figure 11.2 shows the distribution of holes on the property. Down-hole depths range from a few tens of meters to approximately 1,000m. The majority of these holes were either inclined due south or were vertical and tested the Crypto deposit area on north-south sections at a rough average spacing of about 75m.

Table 11.1 Crypto Mineral Resource Drilling Database Summary

Company	Period	Hole Numbers	Core		RC		Total	
			No.	Meters	No.	Meters	No.	Meters
Pinnacle Mines	1958-1959	C1,C2	2	228.6	--	--	2	228.6
Utah International*	1961-1985	CC1 to 46 RC2-4,14,26,35-37	39	16,555.8	8	609.5	47	17,165.3
Cyprus Minerals	1990-1991	CCC1 to15 RCCC-1, 6	17**	9,434.6	2	670.6	19	10,105.2
Lithic Resources	2007-2008	C07-1 to 5 C08-6 to 15	17***	10,638.9	--	--	17	10,638.9
Totals			75	36,857.9	10	1,280.1	85	38,138.0

* *Two Utah holes with prefixes of WW are not included in this table; they are in the database but appear to be water wells and have no assays.*
** *Includes two holes (CCC-6B and CCC-10A) that wedged off existing drill holes.*
*** *Includes holes C08-7A and C08-13A drilled from same sites as holes C08-7 and C08-13, respectively, but at different orientations.*

11.2 Historic Drilling

Pinnacle's holes were drilled by Kissner Drilling Company of Cedaredge, Colorado, and were collared at BX size, reducing to AX size down the hole. The first few of Utah's holes were drilled by Nichols Drilling Company; the remainder, as well as Cyprus' holes, were drilled by Boyles Brothers. Records are incomplete, but it appears that for the both the Utah and Cyprus drilling, core holes were generally initiated at NQ size or equivalent, reducing to BQ size or further as down-hole conditions required. Apparently for Cyprus' Phase I drilling, it was intended that the upper portions of the angle core holes be drilled with rotary drilling, but that was not successful, and core drilling was used instead (Cyprus/Mitsui Joint Venture Geologists, 1990).

Other than a few of the earliest holes, most of the historic drill holes on the property were surveyed down hole, the method varying between operators (Rockingham (2001) and Agnerian (1993) incorrectly reported that the Utah holes had not been surveyed). Utah's gyroscopic data were collected by Mollen-Hauer Surveying Company, and Lithic has copies of their survey reports. Utah's dip-test data were recorded in an internal corporate memo from 1967, now in

Lithic's files. Cyprus' gyroscopic data were collected by Navi-Drill, and Lithic has copies of their survey reports. Lithic personnel were able to locate most of the historic drill collars and surveyed their geographic location using a differential GPS.

Bernardi and Ohlin (1991b) noted that previous holes drilled at inclinations of 55° or less at Crypto tended to shallow, which meant that intercepts could be shallower and further south than originally thought.

MDA has no further details on drilling procedures by previous operators.

11.3 Drilling by Lithic Resources Ltd.

For its 2007-2008 drilling program, Lithic used Connors Drilling LLC of Montrose, Colorado. Lithic's program initially involved one Atlas Copco CS-14 drill. A Longyear 44 drill rig was subsequently added and then replaced with a second CS-14. Once bedrock was reached, all holes were started at HQ size, producing a 63.5mm-diameter core. When necessary to overcome drilling difficulties, a number of the holes were reduced in size to NQ, a 47.625mm-diameter core.

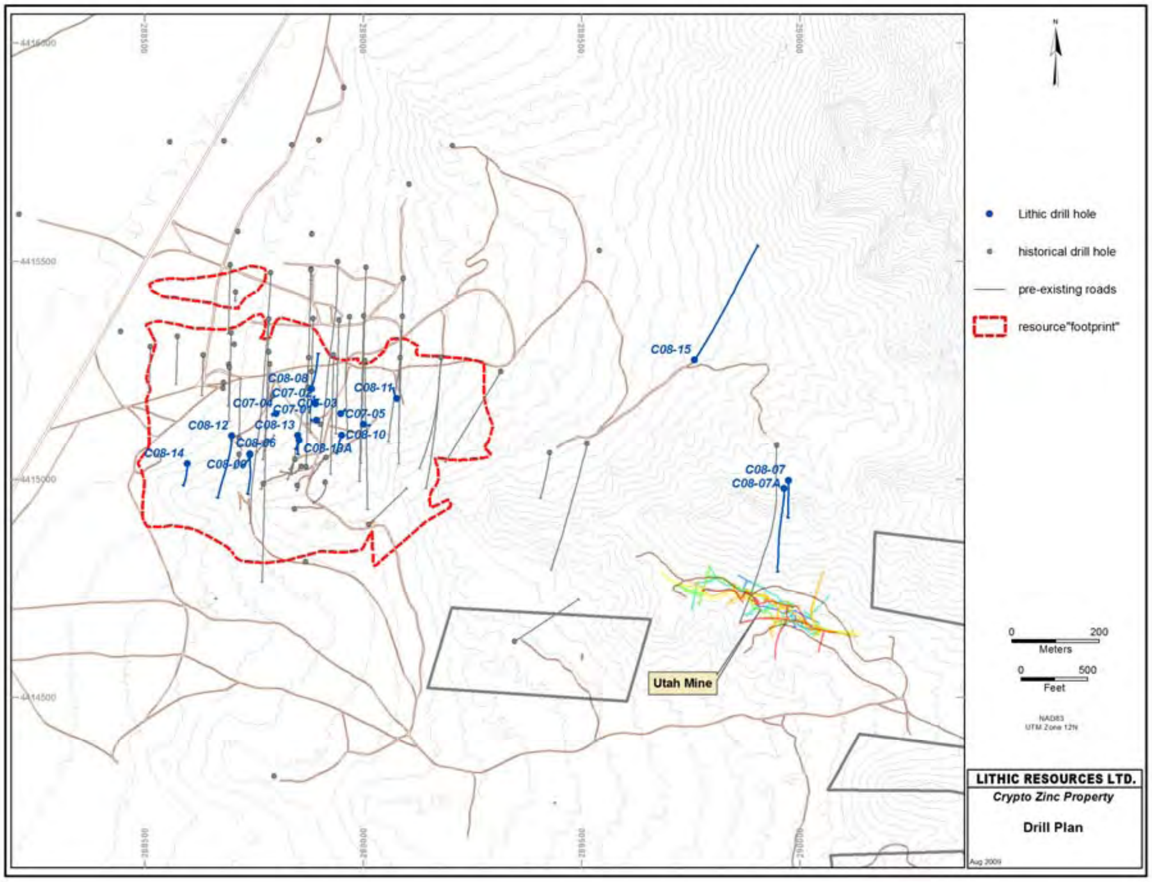
The core was placed in 1.2m-long, three-run wooden core boxes at the drill site. Drillers measured the core recovery as the core was put into the core boxes. Loaded core boxes were then trucked a few hundred meters to Lithic's on-site core logging, sampling, and storage facility.

Once delivered to Lithic's on-site facility, the core was measured for RQD and core recovery, and core markers were converted to metric units. Subsequently, all core was digitally photographed, wet and dry, in a fixed setup to assure constant exposure, distance, and focus, before being geologically logged and marked for sampling.

The down-hole orientations of the drill holes were measured by a technician from International Directional Services using a gyroscopic instrument not affected by magnetic variations. The instrument was lowered to the bottom of each hole, and measurements were digitally recorded every 15.2m (50ft) as it was raised back up the hole. Recorded data included depth, azimuth, inclination, and temperature. Finally, corrected survey data were derived on site and delivered as a digital file to Lithic's on-site staff.

Figure 11.1 Drilling by Lithic on the Crypto Property
(Crypto resource approximately underlies the center of the image.)

Figure 11.2 Location of Drill Holes on the Crypto Property
(Provided by Lithic Resources Ltd., 2009)



12.0 SAMPLING METHOD AND APPROACH

12.1 Sampling and Assaying

12.1.1 Historic Sampling

Utah's samples of core ranged from 0.5 to 5m in length, but the majority were between 1 and 2m. Cyprus' core samples ranged in length from 0.3 to 4m, but the majority were between 1.5 and 2m in length. There are no detailed descriptions of sampling procedures available for either company, but reports of the re-logging of significant quantities of Utah core by Cyprus suggest that Utah had followed the then-current industry standard of splitting the core in half, with one half retained and the other sent to the laboratory. Cyprus used the same standard sampling methodology and, as was standard practice at the time, did not insert quality control samples of their own (pers. comm., M. Bernardi - Cyprus Project Manager). Although no relevant information is available, it should be assumed that Utah also did not insert their own quality control samples.

12.1.2 Lithic Sampling

Samples were selected and marked by Lithic's geologist as the core was being logged. Sample intervals were chosen on the basis of lithology, mineralization, and alteration and ranged in length between a minimum of about 0.5m and a maximum of about 2m. Local employees directly supervised by the geologist then either sawed the core in half using a water-cooled diamond saw or, if the core was fragile, contained potentially soluble minerals, or was otherwise unsuitable for sawing, split it in half using a hydraulic core splitter.

12.2 Core Recovery Determinations

12.2.1 Historic Core Recovery

All of the historic drill campaigns recorded core recovery data and the current Lithic data set includes over 12,600 individual core recovery records. The average core recovery for all of the historic drilling is approximately 85%, with 10% of the drill runs having less than 50% core recovery. There is a general correlation within the data set between core recovery and core diameter with the smaller core (AX and BX) having lower recoveries, on average, than the larger NX core. The great majority of the historic drilling is NX-size core.

12.2.2 Lithic Core Recovery

Core recovery data was noted by the drillers for all 17 Lithic core holes. Lithic field technicians separately collected core recovery data for the upper half of drill hole C07-01 and the full lengths of hole C07-03 and C08-07A through C08-15. The average core recovery for both data sets is greater than 97% with just 2% and 3% of the drill runs, for the driller and Lithic data sets, respectively, having less than 50% core recovery. The generally excellent core recovery lends confidence to the current resource estimate.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 Historic Sampling

Sampling information and analytical data for historic drill holes are only available as recorded in historic drill logs. Although assay certificates were not available for either Utah's or Cyprus' work, it is known that Cyprus' samples were analyzed by ALS Chemex ("Chemex") at their laboratory in Sparks, Nevada (pers.comm, Bernardi, M.) for a suite of elements including Zn, Cu, Pb, Ag, Au, Co, Mn, Mo, Ni, and Fe (Agnerian, 1993).

Drill logs for Pinnacle's two holes included data for Cu, Pb, Zn, Ag, and Au for almost all sampled intervals, as well as sporadic data for Fe, Sn, Mo, and Co. Cyprus' drill logs included assay data for Cu, Pb, Zn, and Ag for all holes, as well as Au, Fe, Cd, and As data for some intervals in some holes. Utah's drill logs included data for Cu, Pb, Zn, and Ag for all holes, as well as sporadic data for Au, Fe, Sn, Sb, Mo, Co, Cd, As, W, Bi, Mn, and Ni.

No information is available regarding details of analytical methods or any security measures that might have been taken with sampling in historic drill campaigns.

There is no information concerning quality assurance/quality control analyses on the historic drill data.

13.2 Sampling by Lithic Resources Ltd.

Following core splitting, samples were placed in fabric bags labelled with a sample number. A tag with the sample number was placed into each bag. Bags were then sealed and stored in a 6m locked storage container of the type used in rail, truck, and ship transport, which had been modified for use in the field. Samples were delivered by Lithic personnel to Chemex's laboratory in Elko, Nevada, on a weekly to bi-weekly basis. At the request of Chemex, samples estimated to contain more than about 10% zinc on the basis of their sphalerite content were segregated and labelled as "high grade" before delivery to the laboratory. A series of "overlimit" protocols was established for Zn, Ag, Cu, Pb, and Mo, in which samples which exceeded the limits of a given analytical technique were automatically scheduled for re-assay with a technique suited for higher-grade material.

Regular samples were dried, weighed, crushed, split, and pulverized at the laboratory in Elko before being analyzed for gold. Analysis was by Chemex's AA23 method of fire assay fusion of a 30g subsample followed by atomic absorption analysis ("FA-AA"). Sample pulps were then sent to Chemex's Vancouver laboratory to be analyzed for Ag, Al, As, Ba, Be, Bi, Ca, Cd, Ce, Co, Cr, Cs, Cu, Fe, Ga, Ge, Hf, In, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, Rb, Re, S, Sb, Sc, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Y, Zn, and Zr using their ME-MS61 method of combined ICP-MS and ICP-AES technology following a four-acid digestion. The upper concentration limit for most metals of interest was 10,000ppm or 1%. Samples exceeding this limit were re-assayed using one of the techniques described below.

Samples that had been segregated as "high grade" were dried, weighed, crushed, split, and pulverized before being analyzed for gold in Elko using Chemex's AA23 30g FA-AA method. Sample pulps were then sent to Chemex's Vancouver laboratory to be analyzed for Ag, As, Bi, Cd, Cu, Mo, Pb, and Zn using their "ore grade" OG-62 method of combined atomic absorption and ICP-AES technology following a four-acid digestion. Regular samples whose metal contents exceeded the limits of the ME-MS61 method were re-assayed by the OG-62 method. Samples that were found to contain greater than 30% zinc using the OG-62 method were re-assayed using Chemex's ME-CON02 method. Indium was not determined in the OG-62 method, and samples were either re-assayed for indium through ME-MS61 if zinc contents were low enough or through a concentrate method such as ME-CON02 or ME-MS61c if they were too high.

13.3 Lithic Quality Control Protocol

A quality-control ("QC") protocol involving a variety of standards and duplicates as well as a blank was implemented by Lithic for all samples from the project. Every 11th sample was either a standard (chosen on the basis of mineralization type and expected grade), a duplicate of some type, or a blank.

A series of commercially available certified standards was chosen and purchased on the basis of expected types and grade ranges of mineralization, including low- and high-zinc values in oxide and sulfide mineralization (Table 13.1). The standard to be used at any given point was chosen on the basis of the mineralization in nearby core. Standards were stored at Chemex's lab and inserted into the sample stream by Chemex personnel as per Lithic's written directions with each shipment. Standards used are listed in Table 13.1.

Table 13.1 Certified Analytical Standards Used in Lithic Drilling Program

Sample	Type	Zn	+/-	Cu	+/-	Pb	+/-	Ag	+/-
GBM 396-10	oxide	10,601 ppm	217	2,897 ppm	48.2	1,018 ppm	22.3	11.6 ppm	0.22
GBM 996-7	oxide	110,344 ppm	1,007.7	23,483 ppm	224.3	38,879 ppm	478.8	125.1 ppm	2.12
CDN-FCM-2	sulfide	1.739 %	0.104	0.756 %	0.046	0.479 %	0.038	73.9 g/t	7.3
CDN-HLHZ	sulfide	7.66 %	0.36	0.76 %	0.03	0.815 %	0.06	101.2 g/t	10.8
CDN-HLLC	sulfide	3.01 %	0.17	1.49 %	0.06	0.29 %	0.03	65.1 g/t	6.7
CDN-HLHC	sulfide	2.35 %	0.11	5.07 %	0.27	0.17 %	0.01	111.0 g/t	8.6

(GBM series: Geostats Pty; CDN series: CDN Resource Laboratories):

Several types of duplicates were used as follows:

preparation duplicate: second, duplicate pulp from the original sample reject was requested

assay duplicate: duplicate analysis of the first pulp from a given sample was requested

field duplicate: second half of core collected in the field

Blank material was collected from an outcrop of barren Tertiary rhyolite about 32km south of the Crypto property and inserted into the sample stream at the Crypto site.

14.0 DATA VERIFICATION

From the perspective of data verification, there are two classes of data in the Crypto database. First in terms of verifiability, there are data generated by Lithic. MDA was able to verify these through a combination of site visits, visual inspection of the core, discussions with the Lithic employees who originated the information, and checking of original sources such as laboratory certificates and data files.

Second in terms of verifiability are the historic data. Most of these exist in paper records that Lithic inherited from prior operators. These paper records are completely plausible, but most are photocopies of the original documents. One deficiency is the lack of original analytical certificates, even as photocopies, though Cyprus' analytical work was carried out at Chemex (now ALS Chemex), and the records are still available in the latter's files. Lithic has made inquiries in the hope of obtaining permission to get these original certificates from Cyprus' successor but has not been successful. The closest thing to an original source for historic assays is entries in photocopies of drill logs.

No historic drill core or sample material exists for visual inspection or verification sampling. Lithic did not drill specifically designated twin holes, but many of their holes targeted similar locations within the deposit. Results generally indicated good correlation between the geology encountered within the historic and Lithic drilling. The depth to the mineralization and the variability commonplace within skarn systems would make any twin program impractical.

MDA has audited the historic database using the available sources, as described in the following sections, and has no reason to suspect that any systematic problems exist. However, MDA cannot state that it has checked the historic database using entirely original sources. Utah International and Cyprus Minerals (now Freeport-McMoran) were major companies involved in exploration and mining, and there are no indications that industry-standard practices and procedures were not followed in their work on the Crypto project. The authors believe that the historic data are suitable for use in a resource estimate.

14.1 Verification of Historic Data

14.1.1 Assay Table

MDA received a copy of the historic assay table in the form of an Excel(TM) spreadsheet containing 4,347 records. MDA checked 1,028 or about 24% of the records against the entries in old drill logs. The fields that MDA checked were Ag (oz), Pb (%), Cu (%), Au (oz) and Zn (%). The error rates that MDA found were as follow:

Element	Ag	Pb	Cu	Au	Zn
Error Rate	0.2 %	0.2 %	0.3 %	0.5 %	0.2 %

The errors that MDA identified would not have had a significant effect on the resource estimate. It is assumed but cannot be proven that assay entry errors in the 76% of the historic assay table that MDA did not check would have a similarly insignificant effect.

14.1.2 Collar Table

Some earlier workers at Crypto located the drill-hole collars on a project-specific grid. Lithic uses a UTM grid system based on NAD83. Lithic identified as many of the old drill-hole sites as it could in the field, in part by using aerial photographic images to locate former sites. In those cases in which it was able to identify the sites, Lithic re-surveyed the locations using a differential GPS. Of 68 holes in the database drilled by former operators, Lithic was able to re-survey 45 collar locations.

MDA had little means at its disposal to verify the locations of old collars. MDA did do a plausibility check, using a geo-registered ortho-photo obtained from Lithic. MDA checked for plausibility by looking at the recorded location of the drill hole on the ortho-photo, checking the image for evidence of old disturbance suggestive of drill sites or drill access roads. It was possible to check 67 of the 68 historic drill holes this way, and the locations of all 67 recorded in the current collar table are plausible. The one hole that could not be checked has a recorded location outside the area of the ortho-photo.

14.1.3 Geologic Data Table

The Crypto geology table contains 4,062 lithologic intervals, 3,124 of which are in historic drill holes. MDA checked these against photocopies of the drill logs. All of the checks that MDA did were done on the historic holes. MDA checked the interval limits of 429 records and found that the database differed from the logs in nine instances, none of which would have a significant effect on the geologic model. MDA checked the lithologic coding of 304 records and found differences in six instances, none of which would have a significant effect on the geologic model. It should be noted that differences between interpretations by successions of workers probably account for some of these differences.

14.1.4 Down-Hole Survey Table

As discussed in Section 11.2, other than a few of the earliest holes, most of the historic drill holes on the property were surveyed down hole, and Lithic has survey reports or other documentation of the results from Utah and Cyprus' drilling.

According to information in a table (Table 14.1) obtained from Lithic, the down-hole orientations of the historic drill holes were obtained by one of three methods:

Table 14.1 Down-Hole Survey Methodology for Historic Drilling

Number of Holes	Down-Hole Orientation Method
11	dip test
43	gyroscopic
4	measured from paper section
29	none (collar only)

Lithic reports that for the Utah holes where dip information is shown as "measured from paper section," Lithic personnel derived the dips by measuring on original Utah sections.

MDA has not verified the down-hole surveys for the historic drill holes.

14.2 Verification of Data Generated by Lithic Resources

Data generated by Lithic are the most verifiable, and MDA checked them using different procedures than those employed for the historic data.

14.2.1 Assay Table

Lithic provided MDA with an assay table for use in modeling. In related work, Dr. Giles Peatfield, P.Eng., a consultant independent of Lithic, compiled an assay table independently of Lithic, for use in evaluating the quality assurance/quality control ("QA/QC") data from the Crypto Project (Section 14.3). Peatfield received the analytical data in digital form, directly from the laboratories. MDA obtained a copy of the assay table compiled by Peatfield and used tools available in Microsoft Access™ database software to compare the Peatfield and Lithic tables. The only significant difference that MDA found was one indium analysis that was in the Peatfield table and was not in the Lithic table. This was resolved.

14.2.2 Collar Table

Lithic's collar table was compiled directly from data transmitted by email from the field, where collar locations were determined using a differential GPS. The Lithic collar table is, in effect, the original. MDA did not independently survey the collar locations.

14.2.3 Geologic Data Table

MDA worked with Lithic in the field during three separate visits and again modeling the geology of the deposit on sections at MDA's office. MDA viewed drill core and otherwise worked closely with Lithic on the geologic model. While the geological interpretations are primarily the product of work by Lithic personnel, MDA gained a high level of confidence in the geological data by working closely with Lithic.

14.2.4 Down-Hole Survey Table

The down-hole orientations of Lithic's drill holes were measured in the field by International Directional Services ("IDS") using a gyroscopic survey instrument. The down-hole survey readings were transmitted from IDS to Lithic's office as digital data files, via email. There is typically one such file for each hole. Lithic compiled its down-hole survey table from those individual data files.

MDA checked Lithic's down-hole survey table by compiling its own down-hole survey table using the individual IDS batch files, obtained from Lithic. MDA did not consider it necessary to request that IDS send the batch files directly to MDA independently of Lithic. MDA used tools available in Microsoft Access™ database software to compare the two down-hole survey tables and found no issues.

14.3 Quality Assurance/Quality Control

14.3.1 General

No QA/QC data are available for the historic drill data and it is not known if past operators conducted any QA/QC analyses.

Section 13.3 describes the QA/QC protocol used for Lithic's core drilling in 2007 and 2008. Every 11^{th} sample was a standard, a blank, or one of three types of duplicates (field, preparation, or assay).

Dr.Giles R. Peatfield, P.Eng., a consulting geologist independent of Lithic, reviewed and analyzed Lithic's QA/QC data (Peatfield, 2009). All assay data were sent directly to Peatfield by Chemex. Peatfield's conclusions are summarized here, and his full report is included as Appendix B.

As described in Section 13.2, Chemex's ME-MS61 ICP analysis method was used for samples with lower metal contents, and the ME-OG62 method was used for "high grade" samples. Gold was analyzed with the Au-AA23 method (FA-AA).

14.3.2 Certified Standards

Sulfide standards with certified values for gold, silver, copper, lead, and zinc were obtained from CDN Resource Laboratories Ltd. ("CDN"). (Peatfield listed three CDN standards, but information provided by Lithic (Table 13.1) indicated there was a fourth CDN standard – CDN-HLHC. Lithic reports that the standard CDN-HLHC was only used once but was inadvertently mislabeled in the dataset used by Peatfield.) Standards from oxidized material with certified values for silver, copper, lead, and zinc were obtained from Geostats Pty. Ltd. ("Geostats"). There were also standards assays for indium, cadmium, gallium, and germanium, although there were no certified values for these elements.

Peatfield concluded from his analysis that the results for the CDN and Geostats standards for silver, copper, lead, and zinc were, in general, acceptable. Although a very small number of results for the standards lay outside acceptable limits, they were not far enough outside to be of serious concern.

The same standards were also analyzed for indium, cadmium, gallium, and germanium. Because there were no certified values for these elements, the results did not address accuracy but rather were a very rough measure of the precision of assays for these elements. Peatfield noted that almost all analyses lay within ±10% of the mean but that the analyses for these four elements were less than optimally precise. He concluded that the levels of these trace elements should be determined in concentrates during metallurgical testing and that specific standards should be in place at that time.

14.3.3 Blanks

Peatfield reported that coarse "blank" samples were included in the 2008 core drilling. He noted that assays to date lead one to question if the assumption that the blank material had negligible metal content was, strictly speaking, true.

Peatfield analyzed results of assays for gold, silver, copper, lead, zinc, molybdenum, indium, cadmium, gallium, and germanium and concluded that, on balance, the results for blanks were not perfect but acceptable, especially for zinc, lead, copper, and indium, which are the metals of main interest.

14.3.4 Duplicate Assays

Field, laboratory preparation, and assay laboratory (the lab made two analyses of a single pulp) duplicate assays were obtained by Lithic for 2007 and 2008 drilling, and Peatfield made correlation plots for all three types of duplicates and both methods of analyses (MS61 and OG62). He concluded that, in general and especially for the metals of principal interest, the duplicate analyses suggested relatively good precision.

14.3.5 QA/QC Summary

Overall Peatfield concluded that the results of the various quality control procedures show that assay data from Lithic's 2007 and 2008 core drilling are, in general terms, acceptable.

Peatfield noted that establishing sampling and QC protocols for the project was an evolving process and that there are still unresolved procedural issues. He recommended that a more rigorous method of inserting control samples be instituted for future drilling, particularly for sample shipments for "high grade" assays, which need to have controls inserted in regular rotation, perhaps with different standard materials. Insertion of control samples for the drilling programs should be keyed to sample shipments, rather than being established beforehand on the basis of drilling intervals.

Peatfield also recommended that for future drill programs, alternate material should be found for blanks because there are some minor concerns about the levels of a few of the analyzed elements in the rhyolite used as blank material for a the 2007-2008 drilling.

15.0 ADJACENT PROPERTIES

MDA is not aware of any relevant work on properties immediately adjacent to Crypto.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Ken Major, P.Eng. is the author of Section 16 and takes responsibility for the information presented in this section.

16.1 Metallurgical Testing by Lithic Resources Ltd.

Based on the results of Lithic's drilling on the Crypto property, the main area of the deposit has recoverable values of copper, zinc, indium, and magnetite. Lithic authorized the completion of a preliminary metallurgical test program at G&T Metallurgical Services Ltd.'s ("G&T") facilities in Kamloops, B.C. to evaluate the main sulfide deposit for recovery of zinc and copper. Samples were provided by Lithic, and compositing instructions were provided by KWM Consulting. Use of the term "ore" in this section is meant to describe well-mineralized rock but does not reflect an economic assessment of the mineralization.

16.2 Metallurgical Sample Composites

Lithic organized for the drill core assay rejects to be delivered to G&T's facilities. The assay rejects had been stored as the individual intervals defined by the drill core program.

The interval assays had been provided to KWM Consulting for evaluation. It was determined that the samples could be composited into samples representing three different ore descriptions. The three ore descriptions were Copper – Zinc ore, High Grade Zinc Ore and Low Grade Zinc Ore. The definitions were based on the following criteria:

- Copper – Zinc Ore: Cu-Zn, copper values greater than 0.40%
- High Grade Zinc Ore: High Zn, zinc values greater than 5.00%
- Low Grade Zinc Ore: Low Zn, zinc values greater than 0.50% and less than 5.00%.

The composites were created based on the weighted values of the intervals. The weighted values were based on the lengths of the intercepts. The resulting interval composites were:

- Cu-Zn: 0.75% Cu, 10.24% Zn
- Hi Zn: 9.53% Zn
- Lo Zn: 3.59% Zn

The weighted intervals were combined by G&T prior to testing. For comparison the assays of theses samples were:

- Cu-Zn: 0.78% Cu, 11.4% Zn
- Hi Zn: 9.45% Zn
- Lo Zn: 3.46% Zn

16.3 G&T Metallurgical Test Program

Information provided by Lithic had identified the samples to be from the sulfide zone. The program developed by G&T focused on the conventional processes to generate independent copper and zinc concentrates. The primary objectives of the test program were to determine the:

- Ore Work Index
- Effect of grind on open circuit rougher flotation
- Recovery of copper to a copper concentrate
- Recovery of zinc to the zinc concentrate
- Grade (%Cu) of the copper concentrate
- Grade (%Zn) of the zinc concentrate
- Deportment of secondary values of indium, gold and silver
- Regrind requirements

The initial testwork focused on the Cu-Zn ore composite. A Bond ball mill work index was completed on this composite to provide background for the preliminary sizing of the grinding equipment. The initial grinding and flotation tests focused on determining the grind requirements to maximize recovery to the respective rougher flotation concentrates.

After determining the target grinds, an open circuit differential flotation program was developed. Chemistry provides for the selective flotation of the copper sulfide mineralization prior to the zinc sulfide mineralization. To minimize the recovery of zinc to the copper flotation concentrate there is a requirement to add zinc mineral depressants. The addition of copper sulfate is required to promote the recovery of zinc to the zinc concentrate.

After determining the target operating parameters based on the Cu-Zn composite, open circuit tests were then completed on the High Zn and Low Zn composites. The early tests identified that the copper in the feed was insufficient to warrant continuing to recover the copper to a copper concentrate. Subsequent tests on the Hi Zn and Lo Zn composites focused on the recovery of Zn to a zinc concentrate.
On completion of the open circuit tests, locked cycle tests were completed on all three composites to determine the metal recovery and the composite grades. Analysis of the concentrates identified the recovery of indium to the zinc concentrate.

The test program also included very preliminary results to evaluate the potential to recover a magnetite concentrate from the skarn deposit. Tests were completed on various process streams to also identify the location for the magnetite recovery circuit in the flowsheet. The initial results indicated that it was possible to recover a magnetic concentrate from various streams. Based on these results it was determined that the best location in the flowsheet was to treat the final tailings. Treating the feed stream or various intermediate stream resulted in a significant loss of copper and zinc to the magnetic concentrate. These would be impurities in the marketability of the magnetic concentrate. The resulting flowsheet to generate a marketable magnetite would be more complex.

16.3.1 Metallurgical Results

A Bond ball mill work index test was completed on the Cu-Zn composite. The test indicated a work index of 13.2 (metric) classifying the ore as moderate hardness.

The initial flotation tests were completed to determine a target primary grind. The association of hardness and grind are required to determine the energy requirements of the grinding circuit. The target grind, P80, of the Cu-Zn ore was determined to be 65 microns. Subsequent testing on the Low Zn and High Zn composites indicated that these ores behaved very similarly to the Cu-Zn ore and similar criteria was adopted. Calibration of the lab grinding mill indicated that similar grinds were attained for the same grinding time. As a result energy requirements are expected to be similar.

Open circuit bench scale testing to evaluate the recovery of copper and zinc determined that the use of zinc depressants, $ZnSO4$ and NaCN, was essential to control the recovery of zinc to the copper concentrate thereby optimizing the performance of the zinc recovery circuit. The open circuit testwork also indicated that rougher concentrate regrinds were essential for both the copper and zinc circuits in order to optimize the concentrate grades and recoveries.

The open circuit testwork provided the preliminary criteria to develop flotation locked cycle test procedures. Locked cycle tests were completed to confirm recoveries and concentrate grades identified by the open circuit tests.

Assays indicate that there are secondary payable minerals that will be recovered to the concentrates including gold, silver and indium. The recovery of these metals is inherent to the flowsheet with the metals following the recoverable sulfides. Deleterious elements were not found to be present in the copper and zinc flotation concentrates at penalty levels. However, it was recommended by G&T that future work should monitor bismuth concentration as it approached threshold limits depending on the smelter accepting the concentrate.

Preliminary metallurgical work has also indicated a potential to recover magnetics (skarn deposit) from the flotation tails.

The metallurgical balances for the locked cycle tests are summarized in Tables 16.1 through 16.3.

Table 16.1 Cu-Zn Composite

Flotation Stream	Cu-Zn Composite	Weight	Assay (percent)			Distribution (percent)		
No.	Product	%	Cu	Zn	Fe	Cu	Zn	Fe
1	Copper Rougher Feed	100.0	0.80	11.0	24.0	100.0	100.0	100.0
10	Copper Concentrate	1.3	31.4	8.36	24.7	49.6	1.0	1.3
20	Zinc Concentrate	20.4	1.24	50.4	9.3	31.5	93.6	7.9
21	Final Tail	78.3	0.19	0.76	27.8	18.9	5.4	90.8

Table 16.2 Low Zn Composite

Flotation Stream	Lo Zn Composite	Weight	Assay (percent or g/t)			Distribution (percent)		
No.	Product	%	Cu	Zn	Fe	Cu	Zn	Fe
1	Zinc Rougher Feed	100.0	0.17	3.63	41.1	100.0	100.0	100.0
10	Zinc Concentrate	5.4	1.91	55.1	7.5	59.8	82.5	1.0
11	Total Tail	94.6	0.07	0.67	43.0	40.2	17.5	99.0

Table 16.3 Hi Zn Composite

Flotation Stream	Hi Zn Composite	Weight	Assay (percent or g/t)			Distribution (percent)		
No.	Product	%	Cu	Zn	Fe	Cu	Zn	Fe
1	Zinc Rougher Feed	100.0	0.22	10.7	21.5	100.0	100.0	100.0
10	Zinc Concentrate	19.2	0.83	53.2	10.2	72.7	95.0	9.0
11	Total Tail	80.8	0.07	0.66	24.2	27.3	5.0	91.0

16.3.2 Metallurgical Projections

The preliminary testwork completed by G&T for the Crypto project has been used to develop the metallurgical projections for the sulfide ore. In developing the projections it has been assumed that the copper recovery circuit will not operate when the mill feed grades are expected to be below 0.4% Cu. With the limited testwork that has been completed, other assumptions are that:

- Zinc recovery projections are linear for zinc feed grades greater than 1.0% Zn.
- Copper recovery to the copper concentrate is 80%.
- Indium (In) recovered to the zinc concentrate

Cu-Zn Ore Projections:

- Copper Concentrate
  - % Cu: 32.0
  - % Zn: 6.0%
  - Recovery Cu: 80%
  - Recovery Zn: 3%

- Zinc Concentrate
  - % Zn: 52.5%
  - Recovery Zn: (1.55 x %Zn(feed) + 75.43)%
  - Recovery In: (-1.93 x %Zn(feed) + 68.1)%

Zn Ore Projections:

- Zinc Concentrate
    - % Zn: 52.5%
    - Recovery Zn: (1.55 x %Zn(feed) + 78.43)%, max 95%
    - Recovery In: (-1.93 x %Zn(feed) + 76.8)%

16.3.3 Opportunities

Exploration drilling has identified significant intercepts of oxide zinc mineralization. Similar to the methodology used for the sulfide mineralization a composite of the intercepts was made to complete some preliminary tests to determine the potential to recover the zinc. Kappes, Cassidy and Associates (KCA) completed a zinc mineral evaluation program and conducted sulfuric acid leaching tests. The acid leaching test results indicated extractions of up to 95% of the zinc from the ore into solution. In addition to dissolution of zinc the leach tests also indicated that 78% of the copper and 37% of the indium were extracted from the ore. There was no work completed to determine the process method and resulting recovery of zinc, copper or indium from the acid solution. There are proven technologies available that will be evaluated in future test programs.

The skarn deposit contains a significant magnetite resource. There is potential to generate magnetite concentrate from the flotation tails but additional work will be required to determine the economics.

16.3.4 Future Work

The completed program has generated preliminary metallurgical results that will provide the criteria and support for additional testwork programs that will be required to satisfy the requirements of prefeasibility study and feasibility study activities. Sample selection for future programs should use new core. The grinding program will influence the dimensions of the core depending on what the other requirements for the core are. The selection of samples should also include the opportunity to analyze geological ore types.

The next stage of metallurgical testing should include programs including:

- Grinding work index – SAG, rod mill, ball mills for identified ore types
- Grind optimization study
- Flotation reagent optimization
- Flotation optimization of secondary minerals
- Regrind optimization
- Column flotation (cleaners) evaluation
- Thickening
- Filtering
- Pilot plant studies (if flowsheet complexities identified)
- Zinc oxide leach testing to determine process design criteria and recovery rates for Zn, Cu and In

16.4 Process Design Considerations

The metallurgical testwork that has been completed by G&T confirms that the sulfide zone of the Crypto project will be amenable to processing using a conventional copper – zinc differential flotation process. The process flowsheet will include crushing and grinding facilities to generate a flotation feed with a nominal P80 of 65 microns.

For ores with sufficient copper to operate an economic recovery process, zinc depressants will be added to the flotation feed slurry to minimize zinc recovery to the copper rougher concentrate. Copper rougher concentrate will be reground to 15 microns prior to three stages of copper cleaner flotation to produce a concentrate grading 32% Cu.

Tailings from the copper flotation circuit will be fed to the zinc flotation circuit. $CuSO_4$ will be added to the slurry to activate the zinc (sphalerite) for flotation. Zinc rougher concentrate will be reground to 35 microns prior to 3 stages of zinc cleaner flotation to produce a concentrate grading 52.5% Zn.

Concentrates from the flotation process will be thickened and filtered to provide dry concentrates that will be shipped to the respective smelters.

16.4.1 Process Design Criteria

The process design criteria has been developed based on the results of the recent testwork completed for Lithic resources at G&T and on the results and descriptions from earlier test programs as identified in the table of the design criteria. Targeted plant throughput has been determined by the preliminary review of the mineral resources.

16.4.2 Process Plant Description

16.4.2.1 Primary Crushing and Ore Storage

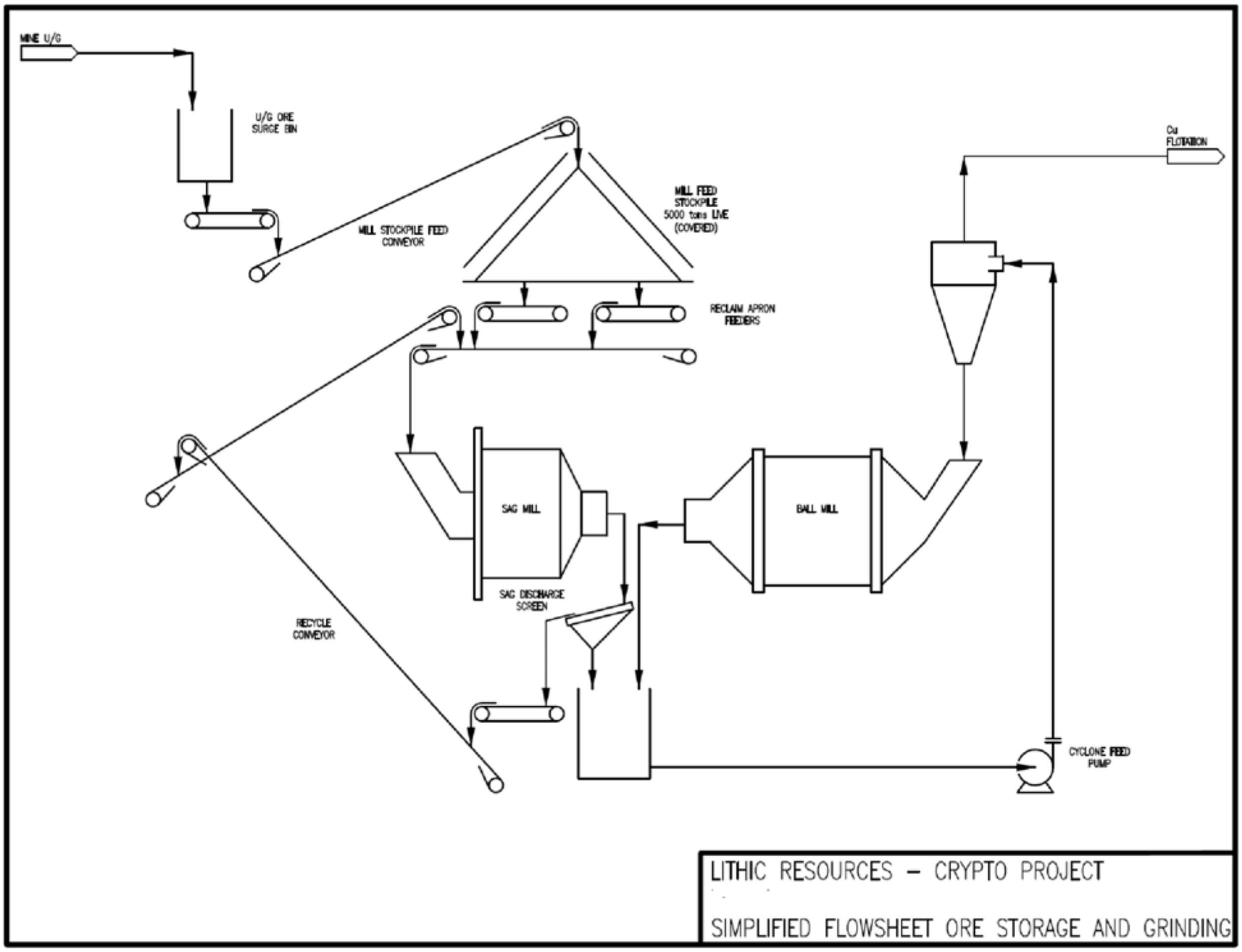
Ore will be delivered from the underground mine ramp to the run-of-mine (ROM) stockpile . The ore will be reclaimed from the ROM stockpile by loader and delivered to the grizzly feeder and primary jaw crusher. Primary crushed ore will be conveyed to the 5,000 ton live mill feed stockpile.

16.4.2.2 Grinding

Two variable speed apron feeders will be installed in a tunnel under the mill stockpile to reclaim the ore to feed the mill. Each apron feeder will have the capacity to deliver 100% of the mill feed.

The apron feeders will discharge to a conveyor that will feed to ore to the grinding circuit. A weightometer will be installed on the feed conveyor to measure and optimize the mill feed. The grinding circuit will be a two stage grinding circuit consisting of a SAG mill and an open circuit ball mill. A crusher has not been included in the SAG grinding circuit because of the high levels of magnetite in the ore and the inability to selectively remove tramp steel from the magnetite in the crusher feed. Figure 16.1 illustrates a simplified flowsheet for ore storage and grinding.

Figure 16.1 Simplified Flowsheet Ore Storage and Grinding



Reclaimed ore will be delivered to a 5490mmØ x 3350mm EGL, 1230kW SAG mill. The slurry discharge from the SAG mill will feed a single deck horizontal vibrating screen. The screen oversize (-50mm, +9mm) will be recycled to the feed of the SAG mill. The recycle conveyors will be 760mm.

The SAG screen undersize will be combined with the ball mill discharge in a common pumpbox and the slurry pumped to a set of ball mill cyclones. The cyclones will be sized to deliver a P80 65micron cyclone overflow product to flotation. The ball mill cyclone underflow will feed to the 4570mmØ x 6100mm, 1865kW ball mill.

Grinding media for the various stages of grinding will be:

- SAG mill 125mm forged
- Ball mill 50mm forged

16.4.2.3 Flotation and Regrind

Grinding cyclone overflow will be piped to the flotation circuit. Recovery of copper and zinc to respective concentrates will be completed using differential flotation techniques with the copper (chalcopyrite) recovered first. Zinc flotation will follow copper flotation with $CuSO_4$ added to promote the flotation of zinc. Due to the additional reagent costs required to operate the copper circuit the PEA has been completed based on the assumption that the copper flotation circuit will be operated when the copper feed grade is above 0.4%Cu.

16.4.2.4 Copper Flotation

Grinding cyclone overflow will be piped to the copper rougher flotation bank (Figure 16.2). A bank of 5, 20 m^3 flotation cells will be used for this application. Rougher concentrate will be pumped to a cyclone to increase the slurry density prior to feeding the copper regrind ISAmill. The ISAmill will use 3mm ceramic grinding media to generate a P80 15 micron grind to feed the cleaner flotation circuit. Copper rougher tailings will be combined with the copper 1st cleaner scavenger tailings and pumped to the zinc flotation circuit.

To generate a marketable concentrate and maintain high cleaner recoveries a 3-stage copper cleaner flotation circuit will be required. The cleaner circuit will consist of:

- 1st cleaner 3, 5 m^3
- 1st cleaner scavenger 3, 5 m^3
- 2nd cleaner 2, 5 m^3
- 3rd cleaner 1, 5 m^3

The intermediate product streams, 2nd cleaner tails and 1st cleaner scavenger concentrate, will be combined with the copper rougher concentrate for feed to the regrind circuit. The depressants NaCN and $ZnSO_4$ will be added to the rougher and cleaners circuits to inhibit the recovery of zinc to the copper concentrate.

16.4.2.5 Zinc Flotation

Copper flotation tails will be piped to the zinc flotation conditioner where $CuSO_4$ will be added to activate the zinc (Figure 16.3). The flotation conditioner will deliver the zinc flotation feed to the zinc rougher flotation bank. A bank of 6, 20 m^3 flotation cells will be used for this application. Rougher concentrate will be pumped to a cyclone to increase the slurry density prior to feeding the zinc regrind ISAmill. The ISAmill will use 3mm ceramic grinding media to generate a P80 36 micron grind to feed the cleaner flotation circuit. Zinc rougher tailings will be combined with the zinc 1st cleaner scavenger tailings and pumped to the final tailings.

To generate a marketable concentrate and maintain high cleaner recoveries a 3-stage zinc cleaner flotation circuit will be required. The cleaner circuit will consist of:

- 1st cleaner 5, 20 m^3
- 1st cleaner scavenger 5, 20 m^3
- 2nd cleaner 5, 20 m^3
- 3rd cleaner 3, 20 m^3

The intermediate product streams, 2nd cleaner tails and 1st cleaner scavenger concentrate, will be combined with the zinc rougher concentrate for feed to the regrind circuit.

Figure 16.2 Simplified Flowsheet Copper Flotation

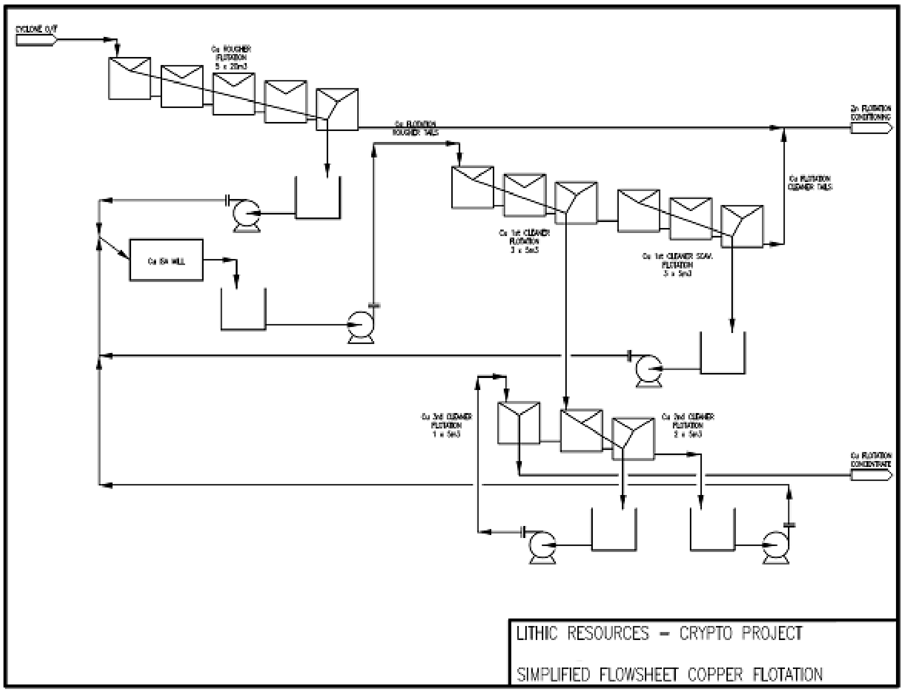
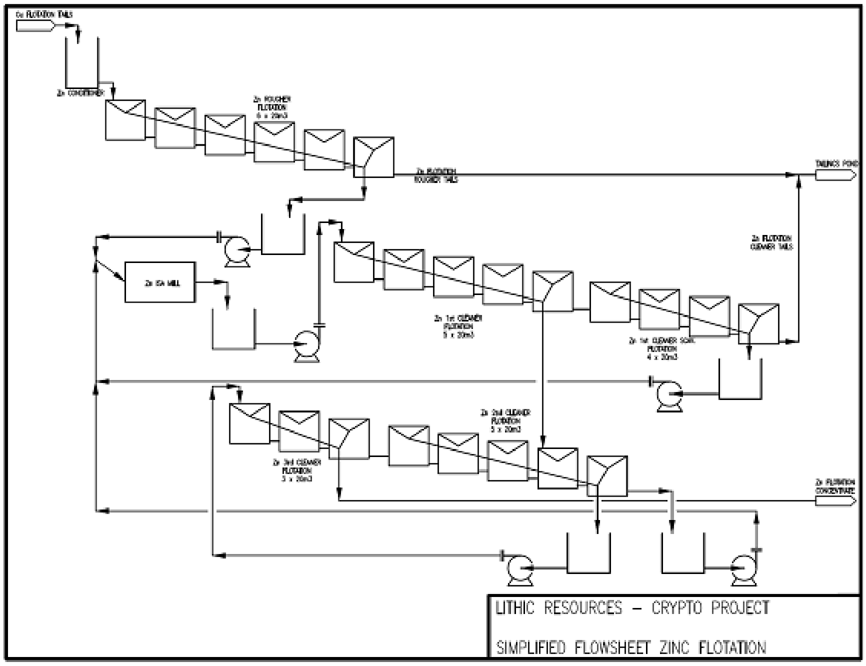


Figure 16.3 Simplified Flowsheet Zinc Flotation



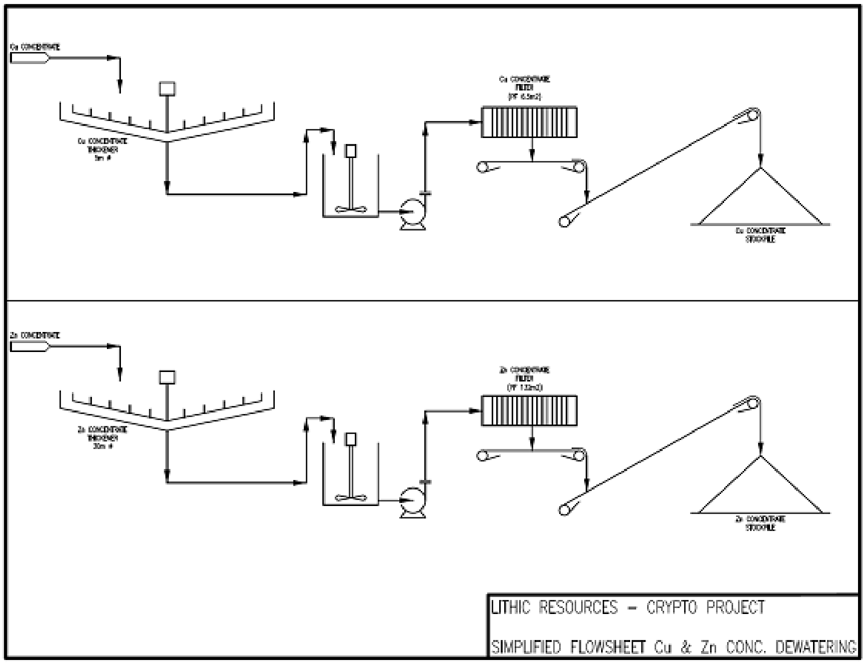
16.4.2.6 Concentrate Dewatering

Copper concentrate will be thickened in a 5 mØ elevated hi-rate thickener (Figure 16.4). Thickener underflow at 60% solids will be pumped to a 6mØx 8m concentrate storage tank providing about 20 hours of concentrate storage ahead of the filter. A 6.3 m^2 Larox pressure filter operating at 90% availability has been selected to reduce the concentrate to 10% moisture for shipping.

Zinc concentrate will be thickened in a 30 mØ elevated high-rate thickener. The thickener will be housed in a pre-engineered structure to protect it from the winter operating conditions. Thickener underflow at 50% solids will be pumped to a 10mØ x 10m concentrate storage tank providing about 20 hours of concentrate storage ahead of the filter. A 132m^2 Larox pressure filter operating at 90% availability has been selected to reduce the concentrate to 10% moisture for shipping.

Filtered concentrate will be stored in an unheated loadout shed. Concentrate will be loaded into bulk trailers using a front end loader. The trailers will be weighed on-site prior to shipping.

Figure 16.4 Simplified Flowsheet for Concentrate Dewatering



16.4.2.7 Grinding Media and Reagents

SAG mill and ball mill grinding media will be delivered on flat bed trucks that will be dumped into bulk storage bins located at the mill. Ceramic media for the regrind mills will be delivered in drums. When bulk delivery is available, reagents will be delivered in large tote bags.

Consumables include:

- 125 mm forged grinding media (SAG)
- 50mm forged grinding media (ball mill)
- 3mm ceramic (copper and zinc regrind mills)
- 3418A (copper flotation collector)
- NaCN (copper flotation zinc depressant)
- $ZnSO_4$ (copper flotation zinc depressant)
- $CuSO_4$ (zinc flotation zinc activator)
- SIPX (zinc flotation collector)
- MIBC (copper and zinc flotation frother)
- Lime (zinc flotation pH modifier)
- Flocculant (copper and zinc concentrate thickeners)

16.4.2.8 Services

Process plant services will include:

- Assay laboratory
- Metallurgical laboratory
- Water storage and distribution systems
- Compressed air supply and distribution system

The assay lab will be a modular self contained building and will include the sample preparation area and sample drying ovens. The metallurgical lab will be installed in the mill building near the mill offices.

Provisions will be made for the installation of fresh water, potable water and process water tanks. The fresh water tank will include a volume allowance for the fire water requirements. Fresh water requirements in the mill required for make-up water based on tailings hold-up will be added to the process at gland water and cooling water addition points and will also be used for reagent mixing.

A process water tank will be installed to provide storage for reclaim water returned from the tailings pond. Water recovered from the copper concentrate thickener and filter will be used for launder dilution requirements in the copper flotation circuit. A similar system will be used in the zinc circuit.

Plant air compressors will be installed to supply general requirements in the mill. Separate compressors will be installed for the clutch air compressors on the SAG mill and the ball mill. The compressors required for the operation of the lead and zinc pressure filters will also be independent of the plant air system.

16.4.2.9 Process Control Philosophy

The mill will be equipped with a sampling system that will provide samples to the on-line analyzer(s) and for the collection of samples for metallurgical accounting. Particle size analyzers will be installed for continuous monitoring of the particle size feeding to various stages of the flotation circuit for the primary grind and each of the regrind streams

The mill will be equipped for measurement and control of reagents and sensing of key operational and equipment parameters with the installation of a DCS control system.

16.5 Process Plant Operating Costs

16.5.1 Summary

A scoping level estimate of the process plant operating costs was prepared in April 2010 US dollars. All of the ore is not amenable to recovery of copper concentrate. The operating costs have been developed for Cu – Zn ore and for Zn ore. Summaries of the unit costs are provided in Tables 16.4 and 16.5.

Table 16.4 Summary of Operating Costs – Cu/Zn Ore

Mineral Processing Operating Costs (Cu/Zn Ore)			
Description	***Annual ($,000)***	***Unit Cost ($/t)***	***Distribution (%)***
Mineral Process Labor	6,012	4.71	22.81
Consumable Supplies	11,960	9.36	45.37
Power	6,412	5.02	24.33
Maintenance Supplies	1.977	1.55	7.50
Sub-Total Plant	**26,359**	**20.63**	**100.00**
5% Contingency	1,318	1.03	
Total	**27,677**	**21.66**	

Table 16.5 Summary of Operating Costs – Zn Ore

Mineral Processing Operating Costs (Zn Ore)			
Description	***Annual ($,000)***	***Unit Cost ($/t)***	***Distribution (%)***
Mineral Process Labor	6,012	4.71	24.62
Consumable Supplies	10,478	8.20	42.90
Power	6,100	4.77	24.98
Maintenance Supplies	1.832	1.43	7.50
Sub-Total Plant	**24,421**	**19.12**	**100.00**
5% Contingency	1,221	0.96	
Total	**25,642**	**20.07**	

16.5.2 Basis for Estimate

Operating costs have been based on processing 1,227,500 tonnes (metric) of ore per annum, at a nominal rate of 3,500 tonnes per day (tpd).

Labor costs are based upon an assumed process plant workforce and labor rates that are similar to current rates in Nevada and Utah.

Reagent costs have been based on consumptions identified by the G&T testwork program and unit costs provided by reagent vendors. Mill liner costs have been based on annual liner replacement for the SAG mill and biannual liner replacement for the ball mill. Grinding media costs have been developed from the abrasion index determined by the G&T testwork and typical consumptions for similar operations. Vendor pricing was obtained for grinding media.

A freight rate of $150 per ton has been applied to reagents and grinding media. Approximately 8,200 ton of reagents and media need to be transported to site. The costs of freight account for about 10% of the operating supply costs.

16.5.3 Labor

The suggested staffing and costs for the mill operations and maintenance crew have been included in Table 16.6 and are based on information received from an operating mine near Elko, Nevada.

Table 16-6 Mill Staffing

Process Plant Operations and Maintenance			
Hourly	***Employees***	***Cost ($,000/yr)***	***Unit Cost ($/t)***
Operations Group	34	2,711	2.12
Maintenance Group	18	1,493	1.17
Salaried			
Operations Group	13	1,383	1.08
Maintenance Group	4	424	0.33
Totals	**69**	**6,012**	**4.71**

16.5.4 Reagents and Grinding Media

The reagent requirements have been estimated based on the addition rates defined in the G&T test program. Typical operating plants operate with less reagents than identified by the test program.

Table 16.7 identifies the flotation reagent and grinding media requirements for the Project. Operating costs have been based on budget quotes received from suppliers.

16.5.1 Power

A preliminary equipment list has been developed for the mill facility. The equipment and respective power requirements have been defined in this list for use in developing plant-site power requirements and unit costs. Estimated installed power for the process facility is 12,600 hp. The operating power has been estimated at 9,150 hp. Power costs have been based on a rate of $0.08 per kWH.

16.5.1.1 Maintenance Costs

Maintenance cost was added as an allowance and has been estimated as 7.5% of the total process costs based on operating plant experience. These costs are meant to cover the costs for materials and supplies used for equipment rebuilds during scheduled and unscheduled maintenance activities.

Table 16.7 Flotation Reagents and Grinding Media (Cu / Zn Flotation)

Consumables	Unit Cost	Cost Units	Usage	Units	Total Cost ($/year)	Unit Cost ($/t)	Cost Distribution
Freight	150.00	$/t	8181	tons	1,227,113	0.96	10.30%
125 mm forged	1275.00	$/t	0.621	kg/t	1,011,515	0.79	8.50%
50 mm forged	1275.00	$/t	0.947	kg/t	1,542,333	1.21	12.90%
3mm ceramic - copper regrind	10.00	$/kg	0.005	kg/t	63,875	0.05	0.50%
3mm ceramic - zinc regrind	10.00	$/kg	0.010	kg/t	127,750	0.50	3.60%
SAG Mill Liners	650000.00	$/set	1.000	annual	650,000	0.36	2.60%
Ball Mill Liners	590000.00	$/set	0.500	annual	295,000	0.16	1.20%
Cu Collector – 3418A	2.90	$/kg	0.060	kg/t	222,285	0.17	1.90%
Zn Collector - SIPX	2.65	$/kg	0.110	kg/t	372,391	0.29	3.10%
Frother - MIBC	2.95	$/kg	0.400	kg/t	1,507,450	1.18	12.60%
Lime	0.35	$/kg	2.750	kg/t	1,229,594	0.96	10.30%
Zn Depressant - NaCN	3.20	$/kg	0.125	kg/t	511,000	0.40	4.30%
Zn Depressant - ZnSO4	1.20	$/kg	0.375	kg/t	574,875	0.45	4.80%
Zn Activator - CuSO4	2.05	$/kg	1.000	kg/t	2,618,875	2.05	21.90%
Flocculant.	3.80	$/kg	0.001	kg/t	3,884	0.00	0.00%
TOTAL CONSUMABLES					**11,957,939**	**9.36**	**100.00%**

16.6 Process Plant Capital Cost

The capital cost for the process plant has been developed based on process equipment costs provided by vendors and factoring these costs to estimate the total capital required.

The costs for site infrastructure facilities have been based on allowances. These costs will be developed during the engineering phase of the project. Site infrastructure includes offices, assay lab, access roads and power line facilities. To determine the installed costs, the equipment costs are factored by 3.5. Installed costs include piping, electrical, foundations, structural steel, chutes and platework. Indirect costs include engineering, construction management, Owner's costs, first fills, capital spares, etc. Indirect costs have been based on 35% of direct costs.

Table 16.8 lists capital cost estimates on an item by item basis.

Table 16.8 Millsite Capital Cost Estimate

CLASSIFICATION		DESCRIPTION	Equipment Cost	Status	Installed Cost
PLANTSITE SERVICES					
	POWER TRANSMISSION LINE	17 MILES		Phone	3,400,000
	SUBSTATION			Allowance	1,500,000
	ACCESS ROAD			Allowance	250,000
	TAILINGS			Estimate	
ANCILLARY BUILDINGS/FACILITIES/FUEL/TRUCKSHOP					
Fresh/Fire/Potable Water Storage & Distribution				Allowance	500,000
	WATER SYSTEMS	POTABLE / FRESH / FIRE			
Assay Laboratory					
	FACILITY	ASSAY LAB		Allowance	1,000,000
Shop Warehouse					
	FACILITY	MECHANICAL SHOP/WAREHOUSE/MINE OFFICE		Allowance	3,500,000
COARSE ORE STOCKPILE AND PRIMARY CRUSHER					
	GRIZZLY FEEDER	RUN-OF-MINE	100,000	Allowance	
	JAW CRUSHER	PRIMARY	1,900,000	Allowance	
	APRON FEEDER	SAG MILL FEED No.1, VFD	82,500	Budget	
	APRON FEEDER	SAG MILL FEED No.2, VFD	82,500	Budget	
	CONVEYOR	COARSE ORE STOCKPILE FEED	310,000	Budget	
	STOCKPILE	COARSE ORE (5000t LIVE)			
	STOCKPILE COVER	COARSE ORE			
	CONVEYOR	SAG MILL FEED	255,000	Budget	
CONCENTRATOR					
Plant Water Distribution					
	PROCESS WATER	TANKS / PUMPS	100,000	Allowance	
Plant Air Distribution					
	COMPRESSED AIR	COMPRESSORS / RECEIVERS	75,000	Allowance	
GRINDING					
	SAG MILL	18 ft dia x 11 ft EGL	4,180,000	Budget	
	SAG DISCH SCREEN		100,000	Budget	
	CRANE	GRINDING AREA, 30t	225,000	Budget	
	PUMP BOX	SAG MILL DISCHARGE	70,000	Budget	
	PUMP	SAG DISCHARGE No.1, VFD	57,500	Budget	
	PUMP	SAG DISCHARGE No.2, VFD	57,500	Budget	
	LINER HANDLER	SAG MILL	375,000	Budget	
	BUSTER	PNEUMATIC BOLT/LINER			
	BALL MILL	15 ft dia x 20 ft	3,745,000	Budget	
	CYCLOPAC	GMax - 26":	90,000	Budget	
	PARTICLE SIZE MONITOR	PSI 300	162,000	Budget	
	ON -STREAM ANALYZER	Courier 6iSL	765,000	Budget	
	PUMP, SUMP		8,500		
COPPER FLOTATION					
	CRANE	FLOTATION AREA, 10t	110,000	Budget	
	CU RGHR FLOT BANK	5 x 20 m3	675,000	Budget	
	1st CU CLNR FLOT BANK	3 X 5 m3	270,000	Budget	
	1st CU CL-SC FLOT BANK	3 x 5 m3	270,000	Budget	
	2nd CU CLNR FLOT BANK	2 x 5 m3	180,000	Budget	
	3rd CU CLNR FLOT BANK	1 x 5 m3	90,000	Budget	
	COPPER REGRIND MILL	ISA MILL	905,000	Budget	
	RGHR CONC PUMP#1, VFD		20,690		
	RGHR CONC PUMP#2, VFD		20,690		
	RGRD DISCH PUMP#1, VFD		28,690		
	RGRD DISCH PUMP#1, VFD		28,690		
	1st CL CONC PUMP#1, VFD		20,690		
	1st CL CONC PUMP#2, VFD		20,690		
	1st SCAV CONC PUMP#1, VFD		20,690		
	1st SCAV CONC PUMP#1, VFD		20,690		
	2nd CL CONC PUMP#1, VFD		20,690		
	2nd CL CONC PUMP#2, VFD		20,690		
	3rd CL CONC PUMP#1, VFD		13,405		
	3rd CL CONC PUMP#2, VFD		13,405		
	1st SCAV TAILS PUMP#1, VFD		14,405		
	1st SCAV TAILS PUMP#2, VFD		14,405		
	2nd CLNR TAILS #1, VFD		13,405		
	2nd CLNR TAILS #2, VFD		13,405		
	CU RGHR TAILS #1, VFD		25,690		
	CU RGHR TAILS #2, VFD		25,690		
	CU RGHR SUMP PUMP		7,500		
	CU CLNR SUMP PUMP		7,500		
	PARTICLE SIZE MONITOR	PSI 500	282,000	Budget	

Table 16.8 Millsite Capital Cost Estimate (cont'd)

ZINC FLOTATION					
ZN RGHR FLOT BANK	6 x 20 m3	780,000	Budget		
1st ZN CLNR FLOT BANK	5 x 20 m3	675,000	Budget		
1st ZN CL-SC FLOT BANK	4x 20 m3	540,000	Budget		
2nd ZN CLNR FLOT BANK	5 x 20 m3	675,000	Budget		
3rd ZN CLNR FLOT BANK	3 x 20 m3	405,000	Budget		
ZINC REGRIND MILL	ISA MILL	905,000	Budget		
RGHR CONC PUMP#1, VFD		26,144			
RGHR CONC PUMP#2, VFD		26,144			
RGRD DISCH PUMP#1, VFD		41,944			
RGRD DISCH PUMP#1, VFD		41,944			
1st CL CONC PUMP#1, VFD		43,245			
1st CL CONC PUMP#2, VFD		43,245			
1st SCAV CONC PUMP#1, VFD		26,144			
1st SCAV CONC PUMP#1, VFD		26,144			
2nd CL CONC PUMP#1, VFD		42,245			
2nd CL CONC PUMP#2, VFD		42,245			
3rd CL CONC PUMP#1, VFD		43,245			
3rd CL CONC PUMP#2, VFD		43,245			
4th CL CONC PUMP#1, VFD		43,245			
4th CL CONC PUMP#2, VFD		43,245			
1st SCAV TAILS PUMP#1, VFD		43,245			
1st SCAV TAILS PUMP#2, VFD		43,245			
2nd CLNR TAILS #1, VFD		26,144			
2nd CLNR TAILS #2, VFD		26,144			
ZN RGHR TAILS #1, VFD		49,195			
ZN RGHR TAILS #2, VFD		49,195			
ZN RGHR SUMP PUMP		7,500			
ZN CLNR SUMP PUMP		7,500			
COPPER CONCENTRATE DEWATERING					
CU CONC THICKENER	5 m Dia.	135,000	Budget		
CU CONC THICK O/F #1		7,802			
CU CONC THICK O/F #2		7,802			
CU CONC THICK U/F #1		7,802			
CU CONC THICK U/F #2		7,802			
CU CONC SUMP PUMP		7,500			
CU FILTER FD PUMP #1		27,390			
CU FILTER FD PUMP #2		27,390			
CU FILTER WATER PUMP					
CU CONC FILTER	PF 6.3/6.3	500,000	Budget		
CU FILTER DRYING AIR COMP		35,000	Budget		
ZINC CONCENTRATE DEWATERING					
ZN CONC THICKENER	30 m Dia.	950,000	Budget		
ZN CONC THICK O/F #1		13,405			
ZN CONC THICK O/F #2		13,405			
ZN CONC THICK U/F #1		13,405			
ZN CONC THICK U/F #2		13,405			
ZN CONC SUMP PUMP		7,500			
ZN FILTER FD PUMP #1		65,445			
ZN FILTER FD PUMP #2		65,445			
ZN FILTER WATER BOOSTER					
ZN CONCENTRATE FILTER	PF132M1	2,368,000	Budget		
ZN PRESSING AIR COMP		35,000	Budget		
ZINC DRYING AIR COMP		65,000	Budget		
SUBTOTAL MILL PROCESS EQUIPMENT COST		**25,032,300**	3.2		**80,103,360**
SUBTOTAL PLANTSITE INFRASTRUCTURE AND MILL					**90,253,360**
INDIRECTS: ENGINEERING, CONSTRUCTION MANAGEMENT, OWNERS COSTS, ETC.			15%		13,538,004
TOTAL: PLANTSITE INFRASTRUCTURE AND PROCESS				**$**	**103,791,364**
CONTINGENCY			0%	$	-
TOTAL:					**103,791,364**

17.0 MINERAL RESOURCE ESTIMATES

This section describes the first NI 43-101-compliant, publicly released mineral resource estimate for the Crypto project. No estimate of mineral reserves has been made for this report.

17.1 Introduction

Lithic requested that MDA update the resource estimates on the Crypto zinc-copper-indium deposit; historic resource estimates are described in Section 6.2. As of October 2009, 85 drill holes totaling 38,138m exist in the Crypto deposit area. The Crypto drill-hole assay database contains 6,503 zinc assays, 6,505 copper assays, and 2,269 indium assays. Other metals are not considered to be economically significant by Lithic and therefore were not estimated.

The resource estimates include the data and analyses resulting from Lithic's 2008 work program at the Crypto project. This program included the completion of 17 diamond drill core holes. The Crypto drilling primarily targeted sulfide-bearing skarn mineralization, both north and south of the Juab fault. The oxide zone was drilled in the upper parts of holes aimed at testing the lower sulfides. All of the Crypto sample data were used in developing the geologic and mineral models, estimating the resources, and determining resource classification. However, the reported resource estimates for Crypto are hampered by having substantially fewer indium analyses than zinc and copper. This has resulted in an excessively harsh limitation to the amount of Indicated material at Crypto.

The work done by MDA for the current resource estimates included assisting Lithic personnel in the geological interpretations. MDA and Lithic worked together on the geologic interpretations on site and with access to logs and core. While MDA constructed the mineral domains, Lithic actively participated in checking and offering suggestions for modification. Those domains were largely defined by the geology, in conjunction with sample assay grades that formed the basis of the resource models. Four models were made:

- lithologic and density,
- zinc,
- copper, and
- indium.

MDA made three site visits and audited the data derived from drilling, while Dr. Giles Peatfield, P.Eng. analyzed QA/QC data (see Section 14.3).

17.2 Resource Classification

MDA classified the Crypto resources in order of increasing geological and quantitative confidence into Inferred and Indicated categories as defined by the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" in 2005 so as to be in compliance with NI 43-101. CIM mineral resource definitions are given below:

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest *which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.*

Inferred Mineral Resource

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

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

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings, and drill holes that are spaced closely enough to confirm both geological and grade continuity.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

17.3 Crypto Resource Estimates

17.3.1 Procedures

Upon completion of the database validation process, MDA constructed 11 unevenly spaced cross sections 25m to 100m apart and looking west at 270°. The sections were spaced to best fit the existing drilling. One set of sections was made for each of zinc, copper, indium, and lithology. Drill-hole information, including rock type and metal grades, along with the topographic surface were plotted on the cross sections.

Quantile plots of zinc (%), copper (%), and indium (g/t) were made to help define the natural populations of metal grades to be modeled on the cross sections. The plots were reviewed with all metals grouped together, but also with the sulfide evaluated separately from the oxide.

Color-coded assays corresponding to population breaks indicated by the quantile plots along with the geological interpretation were used in the creation of distinct mineral domains. These, in turn, were used to control the estimation. The mineral domains as modeled and drawn on the cross sections are not strict "grade shells" but are created using geologic information for defining orientation, geometry, continuity, and contacts in conjunction with the grades. Each of these domains represents a distinct style of mineralization with unique statistical characteristics. These cross sections were sliced to level plan on 3m intervals to coincide with the block-model block size in that direction. The 11 sliced sections were reinterpreted on 279 3m-separated levels, and these were used to code domain percentages into the block model.

17.3.2 Geologic Background

The Crypto zinc-copper-indium deposit occurs as carbonate-rock-hosted, sphalerite and chalcopyrite-bearing skarn mineralization peripheral to the quartz monzonite intrusive basement rock. Stratabound skarn mineralization is preferentially hosted within thin-bedded limestone and minor calcareous siltstone horizons though proximal skarn does form within the more massive carbonate rocks along the intrusive contact. The extent of the mineralization as currently defined is, on average, 830m vertically, 800m along strike, and almost 400m in width in multiple zones. The deposit strikes roughly 270^{o} and dips variably to the north from 40° and 80° but averages 60°.

There are two relatively distinct areas to the Crypto deposit – the Main Zone north of the Juab fault and the Deep Zone at depth south of the Juab fault. Figure 9.1 presents a cross section showing the relationship of these areas. The Main Zone skarn mineralization and enclosing alteration are often dis-continuous due to disruption of favorable bedding horizons by structural complications and intrusive stoping. Observations to date indicate that magnesian skarn is dominant and that massive magnesioferrite/magnetite is very abundant. However, some high-grade zinc and copper mineralization occurs within high-sulfide skarn with limited or no magnesioferrite/magnetite content. The Main Zone does extend to the sub-alluvium surface, where it has been oxidized to a variable depth of between 100m and 200m. The oxide areas have altered metal distributions from the original hypogene mineralization.

The Deep Zone stratabound mineralization is more continuous and stratigraphic correlations are much clearer within the Juab fault footwall. The stratabound mineral horizons, which can be up to 25m thick, dip steeply to the north immediately above the intrusive contact though become less steep with increasing elevation. Significant thickness of mineralization also occurs as sub-horizontal proximal skarn along the contact of the quartz monzonite intrusive. Calcareous skarn predominates within the Deep Zone, although magnesian skarn is common.

Crypto deposit sulfide mineralization consists of coarse-grained, brown to reddish sphalerite with lesser disseminated chalcopyrite, pyrite, and/or pyrrhotite, with the mineral grade, in general, associated with skarn alteration. In the Main Zone, higher-grade mineralization (i.e., increased sphalerite and/or chalcopyrite content) is associated with pervasive magnesian skarn alteration in which all primary textures generally have been destroyed and there is often evidence of multiple pulses of alteration/mineralization. Deep Zone sulfide mineralization generally occurs in massive magnesioferrite/magnetite intercalated with humite ± periclase skarn, while the calcareous skarn consisting of grossularite, Fe-rich diopside and K-feldspar is rarely mineralized. Retrograde alteration, in the form of serpentine or epidote, is common. Decreasing mineralization is generally accompanied by decreasing intensity of skarn development. In the Deep Zone, fracture-fillings of magnesioferrite in dolomitic marble are common in proximity to intercalated layers of massive magnesioferrite/magnetite.

Copper grades are not directly proportional to those of zinc, its distribution typically only partially overlapping with that of zinc in any given drilled interval. However, there appears to be at least a rough increase in the Cu/Zn ratio with proximity to the intrusion, and discrete zones of copper enrichment are definable. Indium is associated with zinc and the petrographic work has indicated that it occurs primarily in the sphalerite lattice. There is not though a direct relationship between zinc grade and indium grade and the highest indium grades tend to occur in mineralization with a zinc grade of between 0.5 and 5%.

17.3.3 Geologic Model

MDA used a combination of lithology, angles to core axes, structural data and logged sulfide percentages to construct a geologic model which formed the basis for the density model used in the estimate. Because the metals behave differently, MDA also modeled zinc, copper, and indium separately. Table 17.1 presents a list of mineral domains and materials defined for this model. While all metals are globally spatially related, they are not necessarily locally spatially related, thereby requiring separate domains for each metal. The three mineral domains of each metal were constructed by MDA but checked for reasonableness by Lithic.

Table 17.1 Coding and Description of the Crypto Geologic Model

Domain Code	Description
101 201 301	Zinc: Low-grade associated with weak skarn Copper: Low-grade associated with weak skarn Indium: Low-grade associated with weak skarn
102 202 302	Zinc: Mid-grade associated with moderate skarn Copper: Mid-grade associated with moderate skarn Indium: Mid-grade associated with moderate skarn
103 203 303	Zinc: High-grade associated with pervasive skarn Copper: High-grade associated with pervasive skarn Indium: High-grade associated with pervasive skarn
99	Unmineralized or discontinuously mineralized country rock outside of the above mineral domains was not modeled due to insufficient assay data

17.3.4 Sample Coding and Compositing

The metal mineral-domain polygons were used to code drill samples. Quantile plots, along with global zone statistics and spatial location of higher grades, were made to assess validity of these domains and to determine capping levels. After these analyses, MDA chose to cap 23 samples. Descriptive statistics of these sample grades by domain are given in Appendix C.

Compositing was done to 3m down-hole lengths, honoring all material-type and mineral-domain boundaries. The 2m by 2m by 3m blocks inside each mineral domain were estimated using only composites from inside their respective domain. Composite descriptive statistics are presented in Table 17.2.

Table 17.2 Descriptive Statistics of Metal Domain Composites

Zinc Composites

Domain	Valid N	Total Length (m)	Median (%)	Mean (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
101	1221	2932.5	0.34	0.65	0.73	1.12	0.01	4.71
102	526	1224.4	3.77	4.03	2.50	0.62	0.21	15.30
103	153	348.8	14.56	16.66	6.53	0.39	5.65	42.74

Copper Composites

Domain	Valid N	Total Length (m)	Median (%)	Mean (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
201	940	2189.9	0.14	0.16	0.09	0.61	0.00	0.65
202	352	769.0	0.45	0.49	0.20	0.41	0.05	2.03
203	136	277.4	1.15	1.46	1.01	0.69	0.12	7.30

Indium Composites

Domain	Valid N	Total Length (m)	Median (g/t)	Mean (g/t)	Std.Dev.	CV	Minimum (g/t)	Maximum (g/t)
301	302	665.3	3.82	4.28	2.73	0.64	0.08	21.80
302	570	1510.9	23.41	30.82	22.00	0.71	3.57	167.39
303	79	321.6	136.74	152.29	57.13	0.38	61.83	490.00

17.3.5 Density

The density values used in the updated resource estimate are based on 427 density measurements collected by Lithic from diamond drill core in the Crypto resource area. The samples were grouped according to sulfide concentration, metal domain, and lithology. Because most of the Crypto core is solid and unfractured, no adjustments were made to the mean grades of the measured data.

The density values assigned to the various lithologies and domains are given in Table 17.3.

Table 17.3 List of Density Values Used in Model

Model Code	Density	LG Zn	MG&HG Zn	LG Cu	MG&HG Cu	LG Cu&Zn	MG&HG Cu&Zn
	g/cm^3	g/cm^3	g/cm^3	g/cm^3	g/cm^3	g/cm^3	g/cm^3
Alluvium	1.80						
Carbonate	2.65	NA	NA	NA	NA	NA	NA
Silicified Rock	2.45	NA	NA	NA	NA	NA	NA
Skarn	2.90	2.94	3.06	3.09	3.11	3.02	3.09
Magnetite	3.85	3.89	3.76	3.83	3.97	3.86	3.87
Rhyolite	2.60	NA	NA	NA	NA	NA	NA
Intrusive	2.60	NA	NA	NA	NA	NA	NA

LG = low-grade; MG = mid-grade; HG = high-grade

17.3.6 Resource Model and Estimation

The Crypto resource block model replicates the relatively complex metal distributions and geometries. Because of the rather contorted geometries, two passes using Inverse Distance techniques were made in the estimate; a long pass to insure filling in all the blocks and a short pass for the Indicated classification. Indium search parameters were particularly long because of the limited amount of analytical data.

Correlograms were made in numerous orientations and with numerous lag lengths but dominantly within the plane of mineralization. Interestingly, the zinc and indium structures are similar. Copper is distinct. But in all cases, the strike continuity is substantially shorter than the dip. The estimation parameters are given in Table 17.4.

Table 17.4 Crypto: Estimation Parameters for Mineral Resources

Description	Parameter
SEARCH ELLIPSOID PARAMETERS: Zinc (Low-Grade)	
Samples: minimum/maximum/maximum per hole (all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	250 / 250 / 50
(high-grade restriction: grade in %Zn and distance in m)	(1.5 / 200)
Second Pass Search (m): major/semimajor/minor	70/ 70 / 15
SEARCH ELLIPSOID PARAMETERS: Zinc (Mid-Grade)	
Samples: minimum/maximum/maximum per hole (all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	200 / 200 / 40
Second Pass Search (m): major/semimajor/minor	70 / 70 / 15
Third Pass Search (m): major/semimajor/minor (Oxide Zone only)	90 / 90 / 15
(high-grade restriction: grade in %Zn and distance in m)	10.0 / 40
SEARCH ELLIPSOID PARAMETERS: Zinc (High-Grade)	
Samples: minimum/maximum/maximum per hole (all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	150 / 150 / 40
Second Pass Search (m): major/semimajor/minor	50 / 50 / 10
SEARCH ELLIPSOID PARAMETERS: Copper (Low-Grade)	
Samples: minimum/maximum/maximum per hole (all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	250 / 250 / 50
Second Pass Search (m): major/semimajor/minor	70/ 70 / 15
SEARCH ELLIPSOID PARAMETERS: Copper (Mid-Grade)	
Samples: minimum/maximum/maximum per hole (all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	200 / 200 / 40
Second Pass Search (m): major/semimajor/minor	70 / 70 / 15
SEARCH ELLIPSOID PARAMETERS: Copper (High-Grade)	
Samples: minimum/maximum/maximum per hole (all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	150 / 150 / 30
Second Pass Search (m): major/semimajor/minor	50 / 50 / 10

Table 17.4 Crypto: Estimation Parameters for Mineral Resources (continued)

Description	Parameter
SEARCH ELLIPSOID PARAMETERS: Indium (Low-Grade)	
Samples: minimum/maximum/maximum per hole all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	300 / 300 / 50
Second Pass Search (m): major/semimajor/minor	70/ 70 / 15
Third Pass Search (m): major/semimajor/minor (Oxide zone only)	90 / 90 / 15
(high-grade restriction: grade in ppm In and distance in m)	(10 / 40)
SEARCH ELLIPSOID PARAMETERS: Indium (Mid-Grade)	
Samples: minimum/maximum/maximum per hole (all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	300 / 300 / 60
(high-grade restriction: grade in ppm In and distance in m)	(20 / 200)
Second Pass Search (m): major/semimajor/minor	70 / 70 / 15
(high-grade restriction: grade in ppm In and distance in m)	(20 / 40)
SEARCH ELLIPSOID PARAMETERS: Indium (High-Grade)	
Samples: minimum/maximum/maximum per hole (all searches)	1 / 8 / 2
Search Bearing/Plunge/Tilt (all searches)	0^o / -60^o / 0^o
First Pass Search (m): major/semimajor/minor	200 / 200 / 40
Second Pass Search (m): major/semimajor/minor	60 / 60 / 20

MDA classified the Crypto resources by a combination of distance to the nearest sample and the number of samples, while at the same time taking into account amount of underlying data and understanding and use of the geology. The criteria for resource classification are given in Table 17.5. There are no Measured resources within the deposit, primarily due to complexity of the mineralization but also due to limited drill data. The maximum distance criteria for Indicated within the Main Zone is less than that used for the Deep Zone due to the greater variability in domain morphology and metal grades. While MDA is confident that the indium zones do continue as modeled, mostly because of their relationship with zinc (the probable location of indium metal), MDA cannot be sure of the grades, and hence the unfortunately small amount of Indicated material. None of these issues deter from the overall confidence in the global project resource, but they do detract from confidence in some of the accuracy which MDA believes is required for Measured and Indicated resources. Without the downgrading for indium, the Indicated zinc and copper material would likely be two times larger than presently reported.

Table 17.5 Criteria for Crypto Resource Classification

Description for All Metals	Criteria
Measured	
None	
Indicated (Main Zone)	
Maximum distance to nearest sample	40m
And	
Minimum number of holes	2
And	
Minimum number of samples	2
Or	
Max. distance to nearest sample	20m
And	
Minimum number of holes	1
And	
Minimum number of samples	2
Indicated (Deep Zone)	
Maximum distance to nearest sample	50m
And	
Minimum number of holes	2
And	
Minimum number of samples	2
Or	
Max. distance to nearest sample	30m
And	
Minimum number of holes	1
And	
Minimum number of samples	2
Inferred	
Those blocks inside the domains that are not classified as Indicated	
Description for final classification	
Measured	
None	
Indicated	
If zinc and indium are Indicated, or 1) zinc is Indicated outside indium zones, or 2) copper is Indicated outside zinc and indium zones	
Inferred	
If either zinc or indium are Inferred or copper is Inferred outside zinc and indium zones	

(note, there are no Measured, Indicated or Inferred resources outside the defined mineral domains)

Because of the requirement that the resource exists "in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction," MDA is reporting the resources at approximate economic cutoff grades that are reasonable for deposits of this nature that will likely be mined by some combination of open-pit and underground methods but mostly underground. As such, some economic considerations were used to determine cutoff grades at which the resource is presented. MDA considered reasonable metal prices and extraction costs and recoveries, albeit in a general sense, and dropping it a bit to account for that material that would become economic using internal cutoffs.

The resource estimate was based on the possibility that a combination of open-pit and underground methods would be optimal. It assumed that all of the sulfide material will be mined by undergound methods, while the near-surface oxide material will be mined by open-pit. As such, mining/milling costs for the estimate were assumed to be $50/tonne for the underground sulfide material and $18/tonne for the open-pit, oxide material. Metal recovery values used for the estimate, based on the metallurgical test results in Section 16, are an approximate weighted average (using total metal values) of 80% for both the sulfide and oxide material.

The Crypto reported resource is summarized in Table 17.6, while the total Crypto resources are tabulated in Table 17.7. The stated resources are fully diluted to 2m by 2m by 3m (vertical) blocks and are tabulated on a zinc-equivalent ("ZnEq") cutoff grade of 1.0% for oxide material expected to be mined by open-pit methods and 3.0% for sulfide material to be mined using underground methods. All material, regardless of which metal is present and which is absent, is tabulated. Because multiple metals exist, but do not on a local scale necessarily co-exist, the ZnEq grade is used for tabulation. Using the individual metal grades of each block, the ZnEq grade is calculated using the following formula:

$$\%ZnEq = \%Zn + (0.0284 * ppm\ In) + (2.5 * \%Cu)$$

This formula is based on prices of US$0.80 per pound zinc, US$2.00 per pound copper, and US$500.00 per kilogram of indium. The metal prices are based on an approximate 3-year historic average calculated in the fall of 2009. No metal recoveries are applied, as this is the *in situ* resource. Typical cross sections through the Crypto block model showing zinc, copper, indium, and zinc equivalent grades are given in

Figure 17.1,

Figure 17.2, Figure 17.3, and Figure 17.4

The lower cut-off used for the oxide material (1.00% ZnEq) reflects the potential for open-pit mining scenarios for this near-surface material. Pit cones developed using the stated metal prices, recoveries, and costs indicate that the great majority of oxide material at the 1.00% ZnEq cut-off can be mined using open-pit methods. The 2m by 2m by 3m block size likely understates the dilution expected from standard open-pit mining methods, but this block size was used to provide the operator the option for evaluating the deposit, either in total or within specific areas, using underground mining methods. For evaluating open-pit methods, a more appropriate larger block size and dilution can be easily achieved by re-blocking.

Table 17.6 Summary Table of Crypto Resources

Crypto Project Reported Resource

Class	Cutoff ZnEq%	Tonnes	ZnEq (%)	Zn (%)	Zn (tonnes)	Zn (lbs)	Cu (%)	Cu (tonnes)	Cu (lbs)	In (g/t)	In (kg)
Oxide - Indicated	1.00	1,114,000	5.48	4.54	50,536	111,413,000	0.263	2,925	6,449,000	10.3	11,500
Sulfide - Indicated	3.00	5,800,000	6.60	4.44	257,709	568,151,000	0.309	17,892	39,446,000	48.8	283,100
Total Indicated		6,914,000	6.42	4.46	308,245	679,564,000	0.301	20,817	45,895,000	42.6	294,600
Oxide - Inferred	1.00	4,644,000	4.45	3.73	173,414	382,312,000	0.165	7,680	16,932,000	12.5	58,300
Sulfide - Inferred	3.00	13,805,000	6.83	4.84	667,714	1,472,057,000	0.372	51,342	113,191,000	37.4	516,400
Total Inferred		18,449,000	6.23	4.56	841,128	1,854,369,000	0.320	59,022	130,123,000	31.2	574,700

Table 17.7 Crypto Total Resource ZnEq Tabulation

Indicated Resources - Oxide

Cutoff ZnEq%	Tonnes	ZnEq (%)	Zn (%)	Zn (tonnes)	Zn (lbs)	Cu (%)	Cu (tonnes)	Cu (lbs)	In (g/t)	In (kg)
0.50	1,657,000	3.92	3.22	53,377	117,677,000	0.191	3,159	6,963,000	7.7	12,700
1.00	**1,114,000**	**5.48**	**4.54**	**50,536**	**111,413,000**	**0.263**	**2,925**	**6,449,000**	**10.3**	**11,500**
1.50	903,000	6.48	5.38	48,542	107,017,000	0.309	2,790	6,150,000	11.7	10,500
2.00	735,000	7.56	6.28	46,155	101,755,000	0.363	2,670	5,887,000	13.2	9,700
2.50	610,000	8.65	7.19	43,873	96,724,000	0.417	2,545	5,610,000	14.6	8,900
3.00	498,000	9.97	8.31	41,427	91,331,000	0.483	2,407	5,305,000	16.0	8,000
3.50	413,000	11.37	9.49	39,154	86,319,000	0.556	2,295	5,059,000	17.4	7,200
4.00	350,000	12.73	10.68	37,402	82,458,000	0.618	2,165	4,774,000	17.9	6,300
4.50	310,000	13.83	11.66	36,158	79,716,000	0.665	2,061	4,543,000	18.0	5,600
5.00	288,000	14.53	12.30	35,415	78,076,000	0.692	1,992	4,391,000	17.8	5,100
6.00	248,000	16.00	13.62	33,754	74,416,000	0.756	1,873	4,129,000	17.2	4,300
7.00	226,000	16.93	14.45	32,618	71,911,000	0.802	1,811	3,993,000	16.8	3,800
8.00	210,000	17.62	15.04	31,636	69,745,000	0.839	1,765	3,892,000	17.1	3,600

Indicated Resources - Sulfide

Cutoff ZnEq%	Tonnes	ZnEq (%)	Zn (%)	Zn (tonnes)	Zn (lbs)	Cu (%)	Cu (tonnes)	Cu (lbs)	In (g/t)	In (kg)
2.00	8,528,000	5.27	3.32	283,221	624,395,000	0.285	24,261	53,487,000	43.6	372,000
2.50	6,974,000	5.95	3.89	271,055	597,573,000	0.297	20,739	45,722,000	46.4	323,800
3.00	**5,800,000**	**6.60**	**4.44**	**257,709**	**568,151,000**	**0.309**	**17,892**	**39,446,000**	**48.8**	**283,100**
3.50	4,906,000	7.21	5.00	245,173	540,513,000	0.314	15,395	33,940,000	50.4	247,200
4.00	4,267,000	7.73	5.46	233,175	514,063,000	0.318	13,586	29,951,000	51.9	221,300
4.50	3,711,000	8.26	5.92	219,552	484,030,000	0.327	12,127	26,736,000	53.7	199,100
5.00	3,207,000	8.81	6.41	205,555	453,172,000	0.334	10,708	23,607,000	55.1	176,600
6.00	2,411,000	9.91	7.43	179,137	394,929,000	0.329	7,934	17,491,000	58.3	140,700
7.00	1,788,000	11.10	8.47	151,385	333,746,000	0.347	6,208	13,686,000	62.2	111,300
8.00	1,250,000	12.68	9.91	123,881	273,111,000	0.370	4,627	10,200,000	64.9	81,100

Inferred Resources - Oxide

Cutoff ZnEq%	Tonnes	ZnEq (%)	Zn (%)	Zn (tonnes)	Zn (lbs)	Cu (%)	Cu (tonnes)	Cu (lbs)	In (g/t)	In (kg)
0.50	6,625,000	3.33	2.74	181,398	399,913,000	0.132	8,725	19,235,000	10.7	71,000
1.00	**4,644,000**	**4.45**	**3.73**	**173,414**	**382,312,000**	**0.165**	**7,680**	**16,932,000**	**12.5**	**58,300**
1.50	3,805,000	5.17	4.38	166,747	367,614,000	0.186	7,073	15,594,000	13.2	50,200
2.00	3,100,000	5.95	5.09	157,793	347,873,000	0.211	6,525	14,384,000	13.5	41,800
2.50	2,513,000	6.81	5.87	147,588	325,376,000	0.241	6,051	13,339,000	13.8	34,700
3.00	2,027,000	7.78	6.76	137,115	302,286,000	0.274	5,560	12,257,000	14.1	28,600
3.50	1,736,000	8.55	7.46	129,572	285,656,000	0.298	5,178	11,416,000	14.2	24,600
4.00	1,472,000	9.41	8.26	121,587	268,054,000	0.325	4,788	10,556,000	14.0	20,700
4.50	1,240,000	10.37	9.13	113,274	249,726,000	0.358	4,441	9,791,000	14.0	17,300
5.00	1,071,000	11.27	9.94	106,390	234,550,000	0.389	4,163	9,178,000	13.8	14,800
6.00	887,000	12.47	11.03	97,869	215,765,000	0.429	3,804	8,387,000	13.8	12,300
7.00	758,000	13.51	11.95	90,618	199,778,000	0.471	3,572	7,876,000	13.8	10,500
8.00	699,000	14.03	12.42	86,788	191,335,000	0.486	3,397	7,488,000	13.9	9,700

Inferred Resources - Sulfide

Cutoff ZnEq%	Tonnes	ZnEq (%)	Zn (%)	Zn (tonnes)	Zn (lbs)	Cu (%)	Cu (tonnes)	Cu (lbs)	In (g/t)	In (kg)
2.00	21,119,000	5.29	3.49	736,446	1,623,586,000	0.334	70,559	155,556,000	34.3	724,000
2.50	16,420,000	6.18	4.26	699,237	1,541,554,000	0.358	58,799	129,630,000	36.0	591,100
3.00	**13,805,000**	**6.83**	**4.84**	**667,714**	**1,472,057,000**	**0.372**	**51,342**	**113,191,000**	**37.4**	**516,400**
3.50	11,820,000	7.43	5.40	638,081	1,406,729,000	0.385	45,473	100,252,000	37.7	446,100
4.00	10,324,000	7.97	5.91	610,274	1,345,424,000	0.397	40,933	90,241,000	37.5	387,300
4.50	9,176,000	8.43	6.33	581,283	1,281,509,000	0.409	37,531	82,742,000	37.9	347,900
5.00	8,141,000	8.90	6.75	549,674	1,211,824,000	0.417	33,972	74,895,000	39.0	317,300
6.00	6,297,000	9.91	7.62	479,601	1,057,338,000	0.443	27,895	61,497,000	41.7	262,300
7.00	4,637,000	11.13	8.68	402,413	887,169,000	0.470	21,789	48,035,000	44.9	208,100
8.00	3,474,000	12.35	9.81	340,995	751,766,000	0.481	16,704	36,827,000	47.1	163,600

Figure 17.1 Crypto Block Model Section 288875 - % Zn

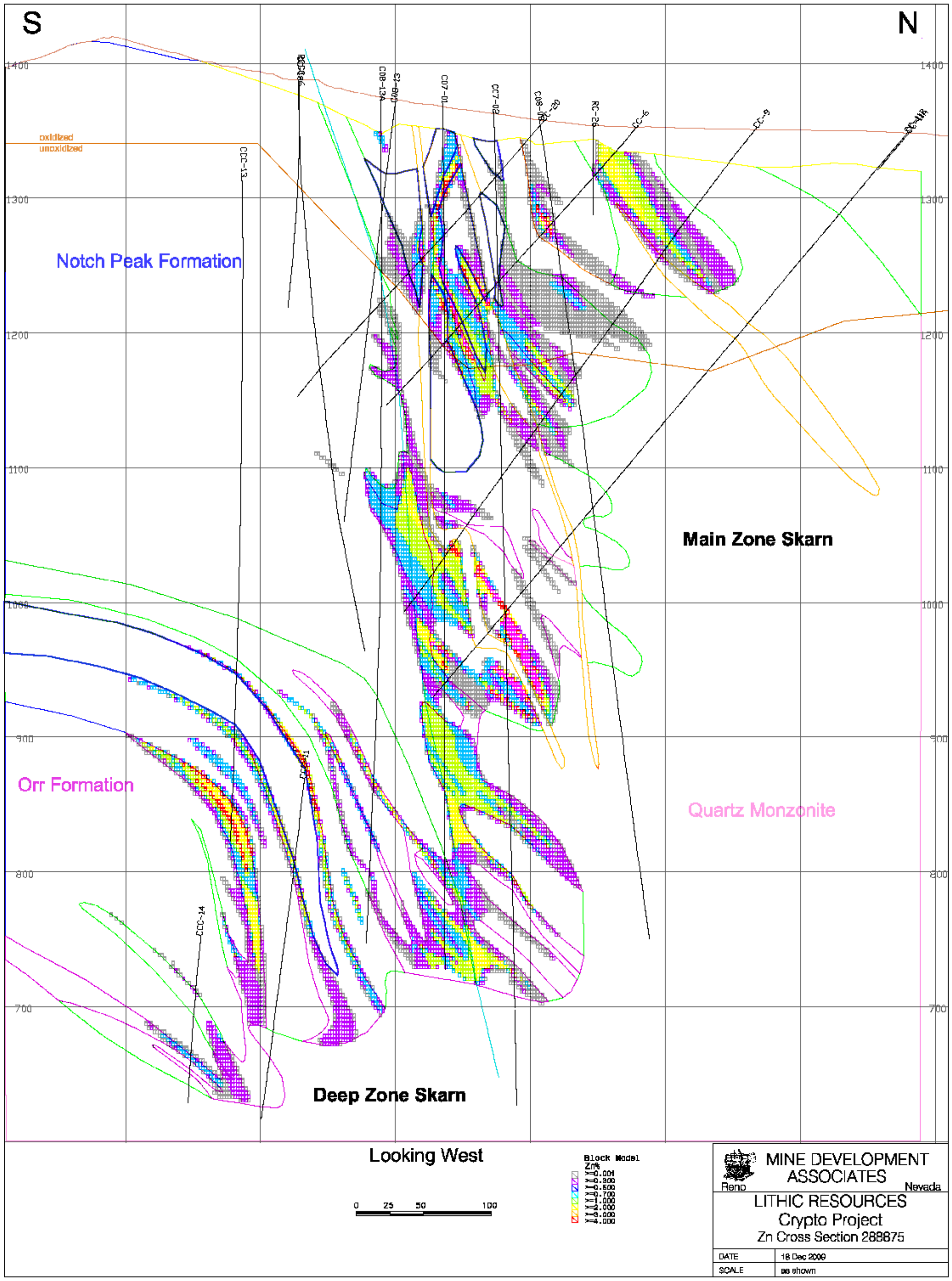


Figure 17.2 Crypto Block Model Section 288875 - % Cu

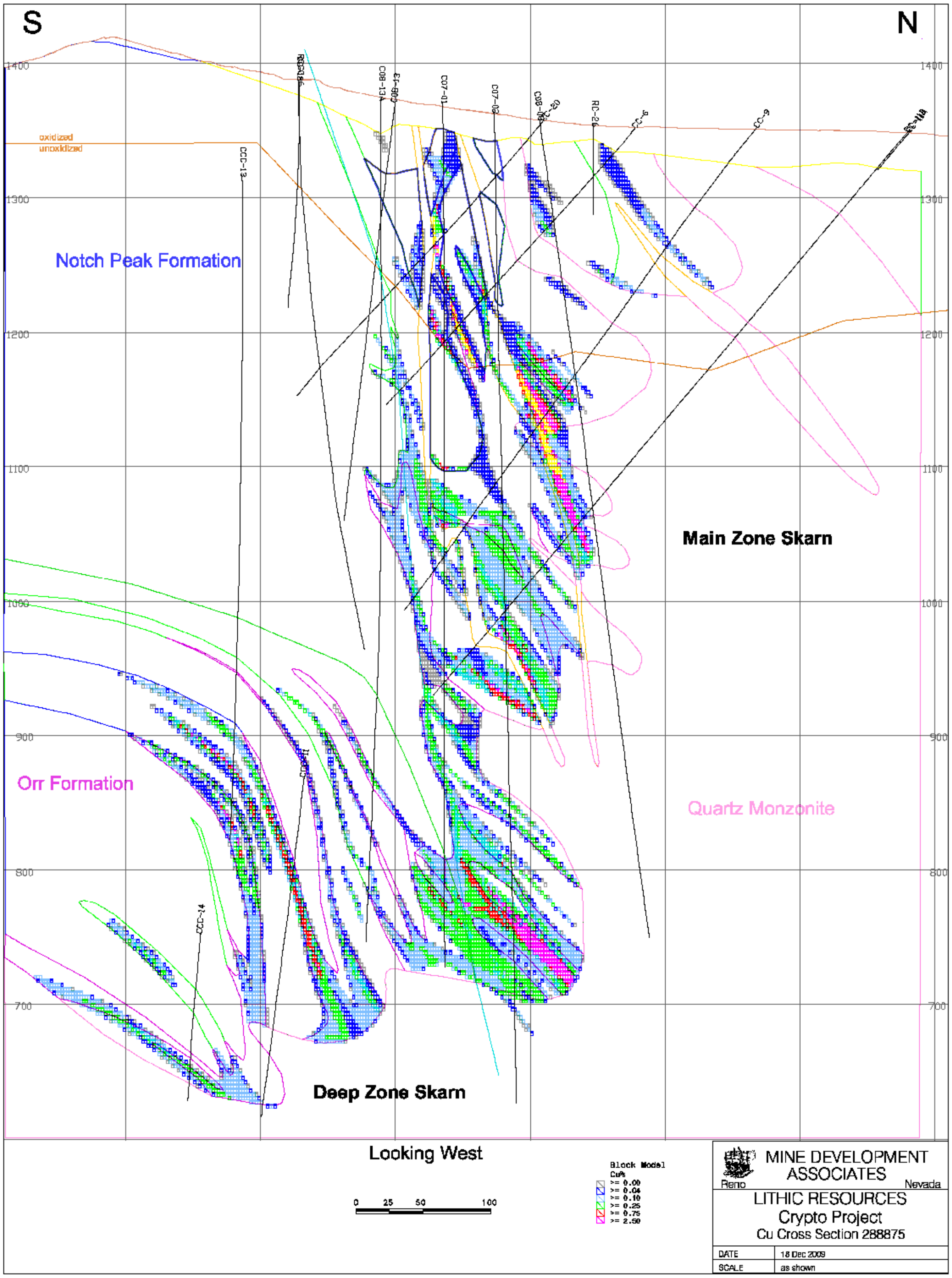


Figure 17.3 Crypto Block Model Section 288875 – ppm In

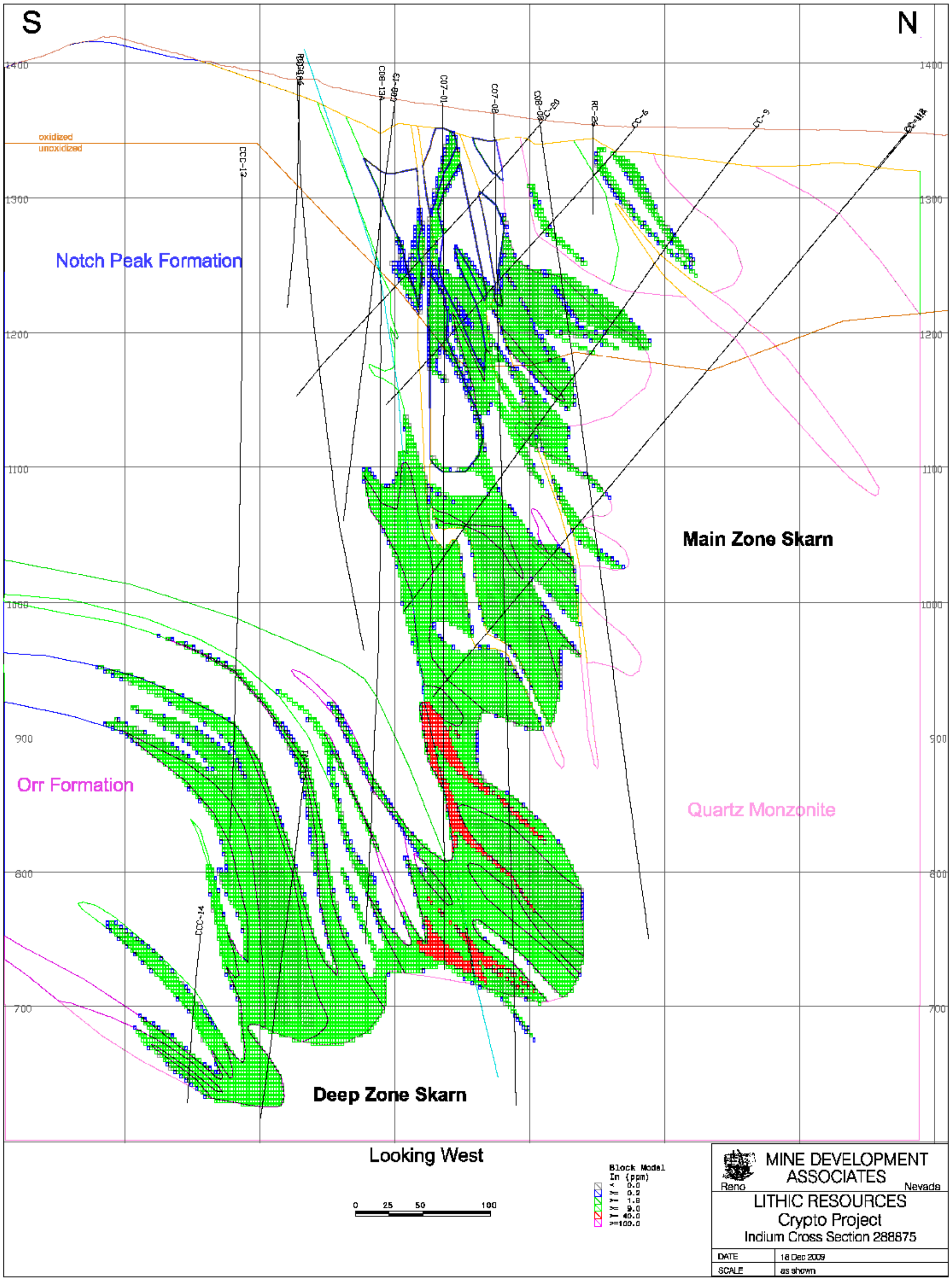
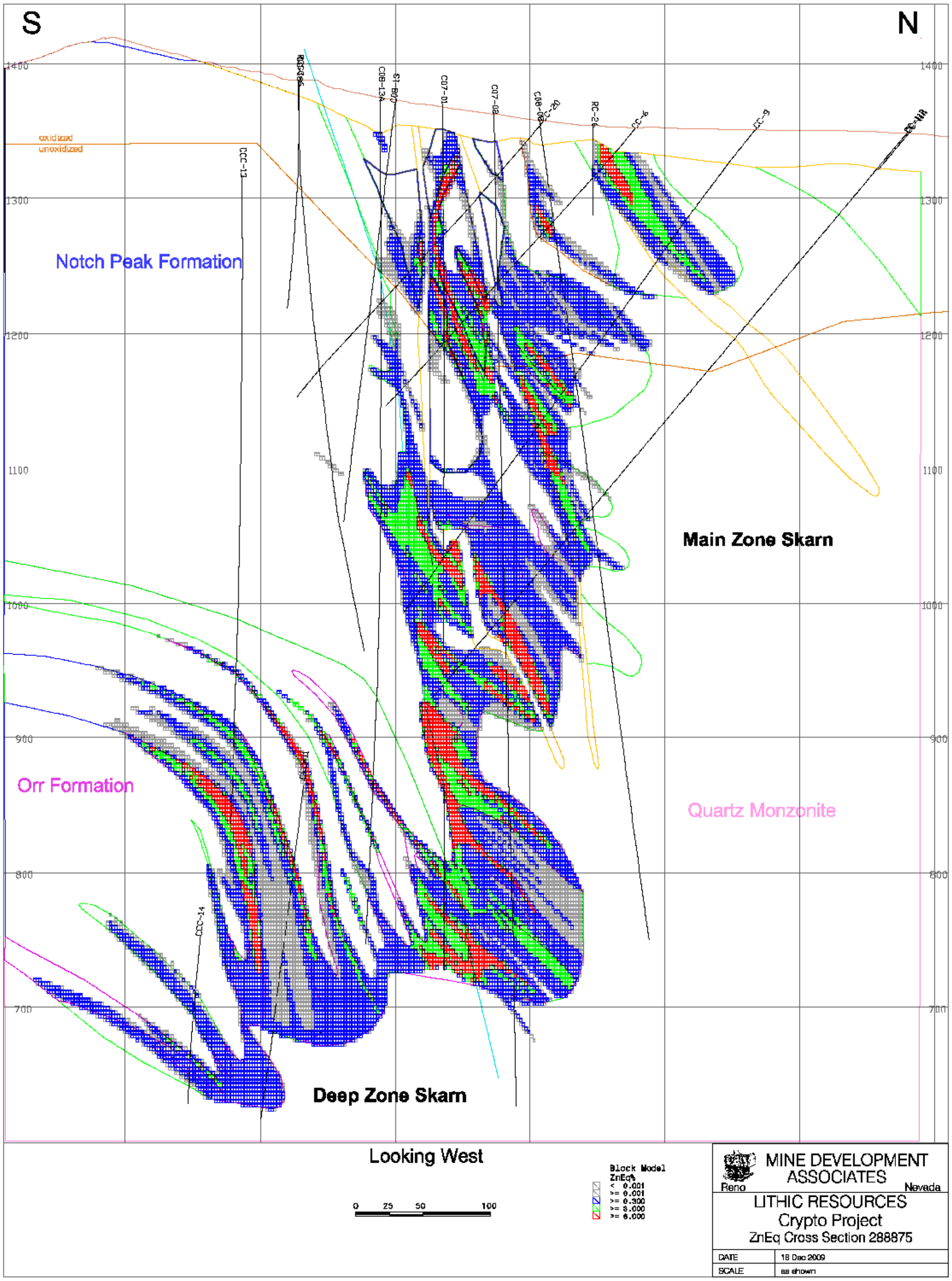


Figure 17.4 Crypto Block Model Section 288875 - % ZnEq



Checks were made on the Crypto resource model in the following manner:

1. Cross sections with the mineral domains, drill-hole assays and geology, topography, sample coding, and block grades with classification were plotted and reviewed for reasonableness;

2. Block-model information, such as coding, number of samples, and classification, were checked visually on the computer by domain and lithology on cross sections and level;

3. Cross-section mineral domain volumes to level mineral domain volumes were checked;
4. Nearest-neighbor and ordinary kriging models were made for comparison;

5. A simple polygonal model was made with the original modeled section domains; and

6. Quantile-quantile plots of assays, composites, and block-model grades were made to evaluate differences in distributions of metals.

In the end, it is deemed that the resource estimates are reasonable, honor the geology, and are supported by the geologic model.

17.3.7 Discussion, Qualifications, Risk, and Recommendations

The detailed work completed by Lithic and MDA on the geologic model, and the data defining the model, has resulted in a resource estimate of high quality. The risk is mostly related to the deposit type. Skarn deposits, such as Crypto, often have relatively complicated and rapidly changing grades and geology. This downside would mostly be alleviated through additional drilling. The upside is clearly dominated by the ability to increase the amount of higher classification material with an increase in indium sample assays. The relatively small amount of Indicated material within the current resource is due to the many fewer samples with indium grades. An increase in indium sample assays would also result in a likely increase in overall indium grade if the tenor of the new sample assays was similar to the existing assays. The current high-grade search restrictions used in the resource estimate serve to constrain the higher grade values resulting in lower indium grades within the Inferred material. Upgrading the classification of material with additional samples would therefore also likely increase the indium grade of this same material. There is also good potential to increase the size of the deposit by targeting extensions of mineralization primarily to the east, west and south.

18.0 MINERAL RESERVES ESTIMATE

No Mineral Reserves have been estimated for this report.

19.0 OTHER RELEVANT DATA AND INFORMATION

19.1 Underground Mining

19.1.1 Introduction

This report describes a proposed mine plan for the Crypto Project. It includes cost estimates, development and mining schedules and a production forecast for underground mining. Data sources include:

1. *Crypto Zinc Project Prefeasibility Study - Capital and Operating Costs*, an internal report for Cyprus Minerals Company (Cyprus, 1991),

2. *Technical Report on the Crypto Zinc-Copper-Indium Project, Juab County, Utah,* an independent report by Mine Development Associates (Tietz et al, 2010).

The mine plan is based on indicated and 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 assessment based on these mineral resources will be realized.

This report is considered suitable for inclusion in a preliminary economic assessment as defined by National Instrument 43-101. The mine plan assumes parameters considered reasonable for mine performance and costs for the deposit described in the references. The author has not visited the site.

19.1.2 Resources and Mineable Material

19.1.2.1 Introduction

The Crypto deposit consists of multiple sub parallel mineralized zones, dipping between 40º and 80º to the north and striking about 270°. The deposit is intersected by the steeply dipping Juab Fault. Two distinct areas have been identified; the Main Zone and the Deep Zone to the north and south of the fault respectively. The following descriptions are taken from Tietz et al (2010):

- "*The Main Zone skarn mineralization and enclosing alteration are often discontinuous due to disruption of favourable bedding horizons by structural complications and intrusive stoping*"
- "*The Deep Zone stratabound mineralization is more continuous and stratigraphic correlations are much clearer within the Juab fault footwall*"

The upper part of the deposit, within the Main Zone, is oxidized to between 100 m and 200 m below the topographic surface at about elevation 1,350 m. This assessment considers the mining of sulfide material between elevations 1,220 m and 710 m. The zones of material above the mining cut off grade extend over strike lengths up to about 600 m and occur within a horizontal width of between about 200 and 300 m. In this area the mineable zones generally dip greater than 60º and vary in horizontal thickness from a few to about 15 m.

19.1.2.2 Cut Off Grade

The preliminary evaluation is based on an in-situ breakeven mining cut off grade of 4.0% zinc equivalent for underground mining of copper/zinc ore and 3.6% zinc equivalent for underground mining of zinc ore. Net block values were calculated for in-situ grades and reported by level. These resources were further adjusted for mining extraction, dilution and recovery as described below. Some of the assumptions used in the cut off grade calculations were revised during the course of the study and were incorporated in the financial analysis. The key metal price and site operating cost assumptions used in the determination of the initial cut off grade are shown in Table 19.1. Metallurgical recovery assumptions are provided in the section Metallurgical Projections.

Table 19.1 Cut-off Grade Parameters

Parameter	unit	
Metal prices		
Zinc	US$/lb	2.00
Copper	US$/lb	1.10
Gold	US$/oz	850.00
Silver	US$/oz	12.00
Indium	US$/kg	500.00
Operating Costs		
General and Administration	$/tonne	5.00
Mining	$/tonne	25.00
Processing Copper Zinc Ore	$/tonne	25.02
Processing Zinc Ore	$/tonne	20.17
Tailings disposal	$/tonne	0.50
Sustaining capital	$/tonne	1.00
Cutoff Grade - Breakeven		
Copper Zinc Ore	$/tonne	56.52
Recoverable Zinc Equivalent	%	3.69
Zinc Equivalent	%	3.97
Zinc Ore	$/tonne	51.67
Recoverable Zinc Equivalent	%	3.38
Zinc Equivalent	%	3.63

19.1.3 Mining Extraction, Dilution and Recovery

Based on the geometry of the mineable zones the use of longitudinal retreat stoping with uncemented rock fill is assumed for all areas. Reasons for this choice are discussed later. For zones greater than about 70 m in strike length it is assumed that 10 m pillars will be left between 70 m long stopes to improve ground stability. Including the material mined during stope drift and slot raise development, this results in an extraction of approximately 88%.

Mining dilution has been estimated based on assumed overbreak of the stope hanging wall and footwall, assumed removal of some fill from stope floors during mucking, and an allowance for variability in the “ore/waste” contact on the stope hanging wall and footwall. The overall stoping dilution is estimated as 16%. It varies with the width of the stope. Because mining limits will be based on assays and in most areas there appears to be lower grade material enveloping the mining resource the grade of the diluting material is nominally set at 10% of the grade of the stope block resource. Note that dilution is defined as the mass of the diluting material as a percentage of the combined mass of the diluting material and above cut-off grade material (W / (O+W) x 100%).

Ore drifts are assumed to have a minimum width of 5.0 m. In some relatively narrow zones this results in wall rock dilution. Overall the dilution for ore drifting is 6%. No dilution is applied to the resource mined during development of the stope slot raises as it is assumed that they will be developed completely within above cut off grade material.

Factors affecting stoping recovery include drilling, blasting and mucking performance. These depend on the drill and blast design and the skill of the operators. Losses can be due to undefined irregular ore outlines, not breaking material included in the design, the inefficiencies of remote LHD mucking and the burial of broken material by falls of ground from the stope walls. An overall loss of 10% of the material broken in the stopes is assumed. For both drift and raise development the recovery of the material broken is assumed to be 100%.

The overall mining extraction, dilution and recovery for ore development and stoping are shown in Table 19.2.

Table 19.2 Mining Extraction, Dilution and Recovery

Source	Extraction	Dilution	Recovery
Ore drift	100%	6%	100%
Slot raise	100%	-	100%
Longitudinal stope	87%	16%	90%
Overall	88%	14%	91%

19.1.4 Resources and Mineable Material

Summaries of the mineral resources and mineable material are shown in Table 19-3. After deducting minor quantities above and below the planned mining areas the total mineral resource considered for underground mining is 12.92 Mt at overall grades of 5.97% Zn, 0.44% Cu and 40.2 g/t In. This includes material classified as Inferred Mineral Resources. After applying extraction, dilution and recovery factors the total mineable material is 12.14 Mt grades at overall grades of 5.19% Zn, 0.38% Cu and 35.0 g/t In as tabulated in Table 19.3.

Table 19.3 Resources and Mineable Material

	Tonnes (millions)	Grade		
		% Zn	% Cu	g/t In
Resources				
Total above breakeven cut off grade	13.34	5.94	0.44	41.1
Less that above 1220 and below 710 elevations	0.42	5.14	0.38	69.0
Included in the mine plan	12.92	5.97	0.44	40.2
Recoverable diluted material				
In stope drifts	1.24	5.66	0.44	37.7
In slot raises	0.23	5.95	0.46	39.7
From stopes	10.67	5.11	0.38	34.6
Total	12.14	5.19	0.38	35.0

19.1.5 Mine Plan

19.1.5.1 Introduction

Primary access for mining is assumed to be by two ramp systems developed from surface. Bored raises are used as ventilation airways and as passes for stope rock fill.

The deposit is characterized by multiple sub parallel zones of material above cut off grade with varying widths and lateral and vertical extents. They are generally steeply dipping. Mining is assumed to be by longitudinal stoping with stope heights of 30 m and strike lengths of up to 70 m with 10 m long strike pillars between stopes. The stopes will be filled with uncemented rock fill.

There are no data on geotechnical or hydrological conditions for the deposit and host rocks. The mining concepts and costs assume excavation sizes, ground support and mine dewatering systems considered reasonable for underground mining in the geological environments described in the references.

19.1.5.2 Access and Service Development

The currently identified material above the mining cut off grade and included in the mine plan extends to about 690 m below surface. The mine plan includes underground access through two ramps from surface (Figure 19.1).

The ramps will be 5.0 m wide and 5.0 m high and located in the footwall of the deposit. Both will provide access to connecting main levels and to ventilation and rock fill raises. One will provide access to each stoping level developed at 30 m vertical intervals from 1190 to 710 m levels.

Raises will be bored between the 710 m level and surface to provide the primary ventilation routes, one for intake and one for exhaust. A raise between the 740 m level and surface will provide a pass for stope rock fill. The raises will be bored in three stages and each will be 3.0 m in diameter.

The rock fill raise will be equipped with truck chutes on the 1130, 920 and 740 m levels with control chains on the 1130 and 920 m levels.

Estimates of lateral development include allowances for regularly spaced cut outs for development waste remuck storage, temporary drainage sumps and safety bays. Other service development will include explosives magazines, materials storage areas, personnel refuge stations, dewatering sumps and pump rooms, and electrical sub stations.

19.1.5.3 Level Development and Mineralization Outline Drilling

Figures 19.2, 19.3 and 19.4 show the outlines of mineable material at the 1130, 920 and 710 m levels based on the current interpretation of core drilling from surface. Note that the outlines in these and the other figures show the unsmoothed shapes from the geologic block model. They represent the approximate strike lengths and horizontal widths of the material above the cut off grade and are suitable for preliminary estimation of development and stoping requirements.

These outlines illustrate the apparent variability and complexity of the geometry of the material above cut off grade. Better definition will be necessary for the design and layout of stoping areas.

Levels will be developed from the main access ramp with the initial lateral development providing connections to the ventilation and rock fill raises.

A cross cut will be developed for the approximate N-S extent of the mining area and drill drifts developed east and west from the cross cut at about 100 m spacings (Figure 19-5). The number of drill drifts will depend on the overall width of the area. The cross cuts and drifts will be 5.0 m wide and 5.0 m high.

On alternate levels rings of core holes will be drilled at 20 m intervals along the drill drift to intersect the mineralized zones from the drift elevation to about 60 m above the level (Figures 19.5 and 19.6). There will be an average of six holes per ring with a total drilled length of about 550 m. The results of the core drilling will be used to design and layout the stopes. This together with sampling results from the stope ore drifts will be used during the design of the stope production blast hole patterns.

Figure 19.1 Schematic Mine Cross Section

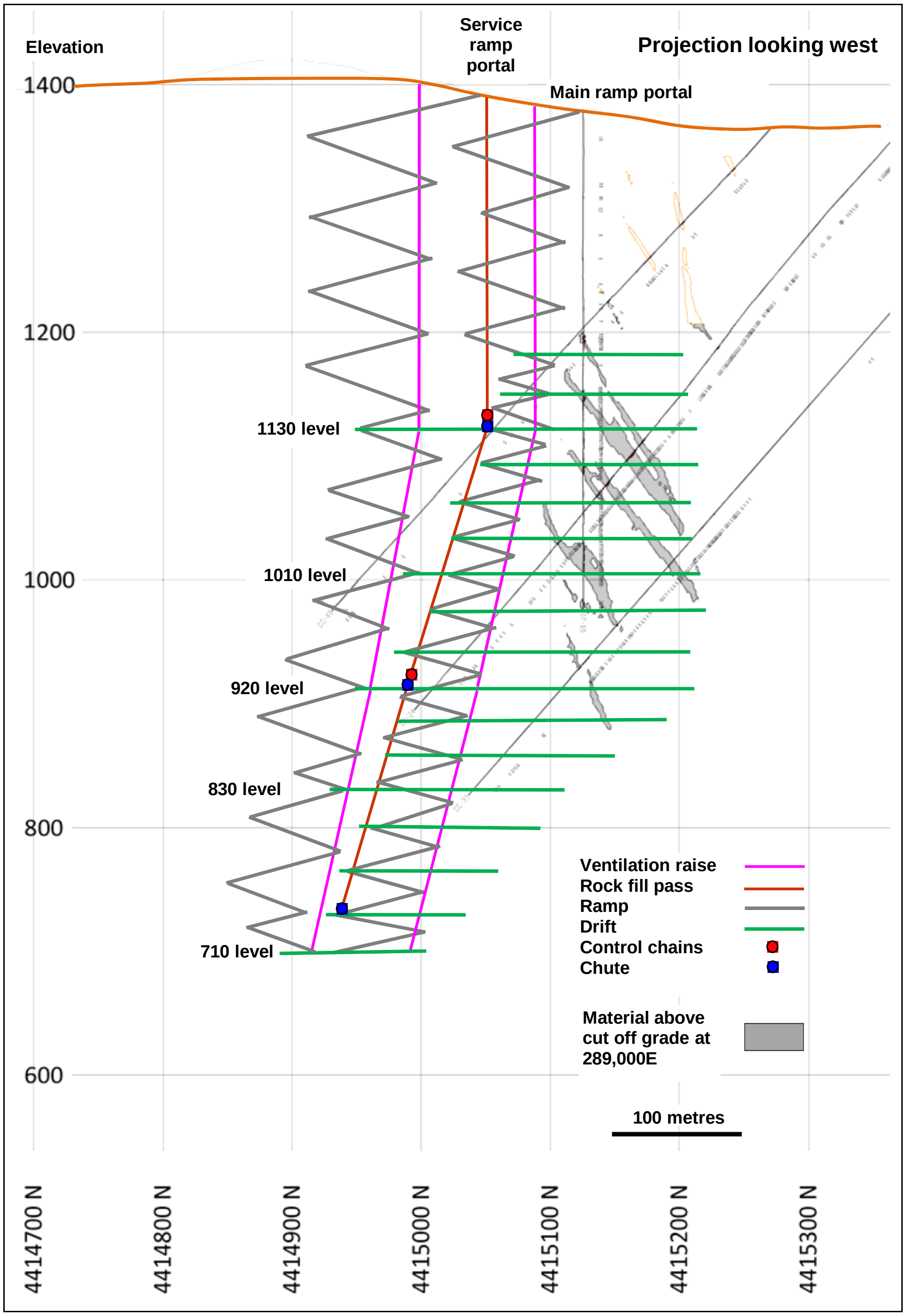


Figure 19.2 1130 Level Plan

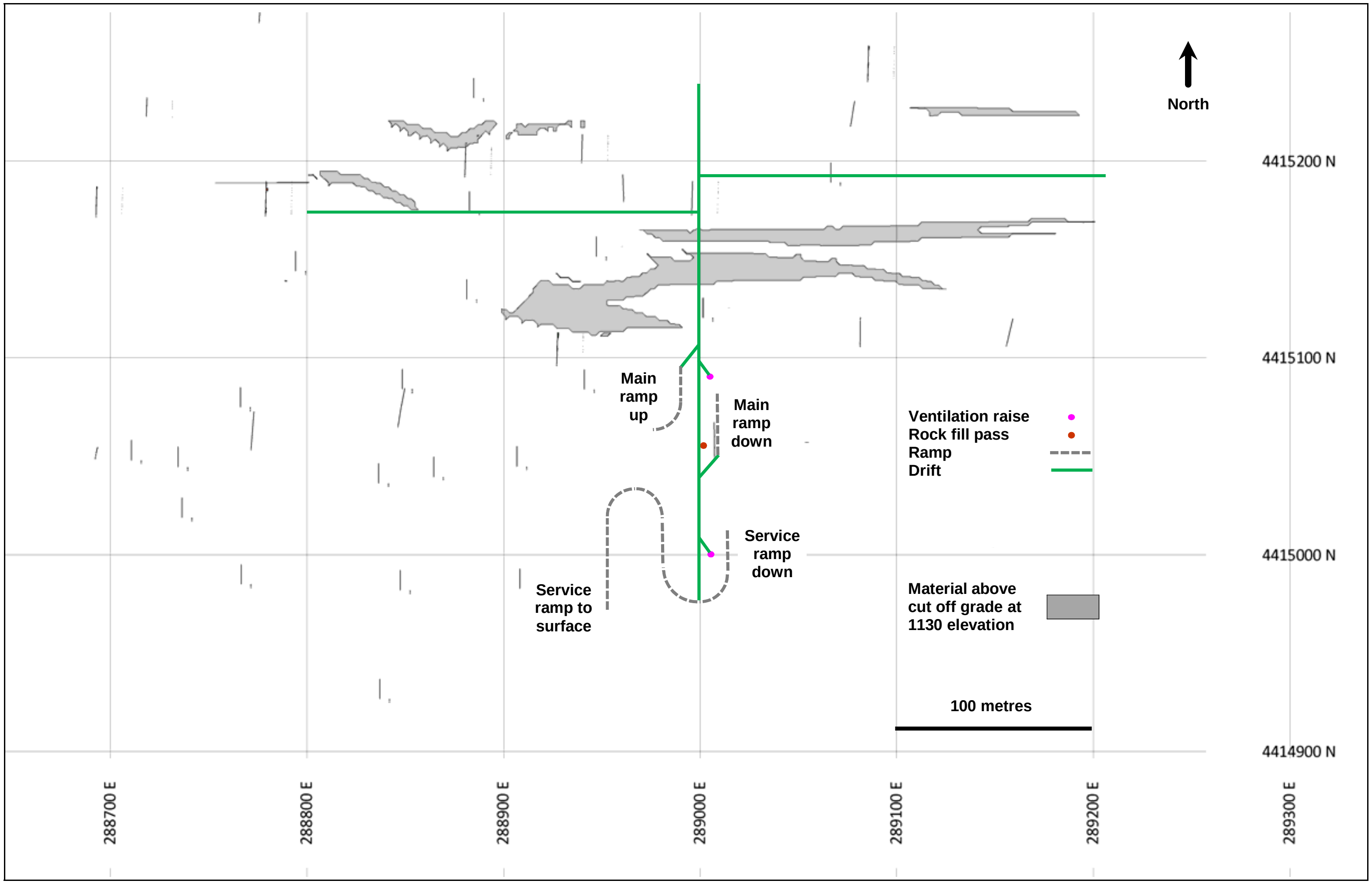


Figure 19.3 920 Level Plan

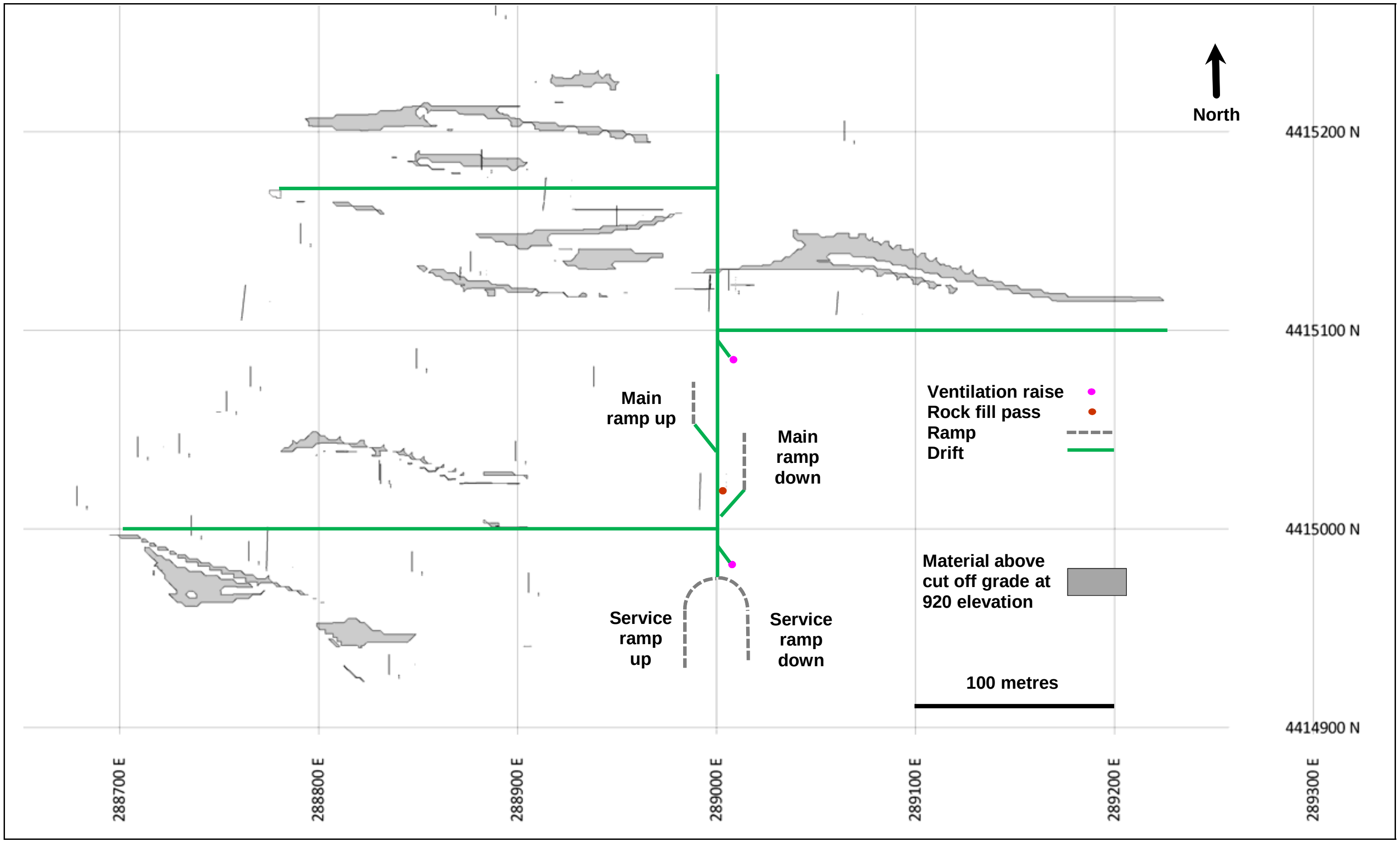


Figure 19.4 710 Level Plan

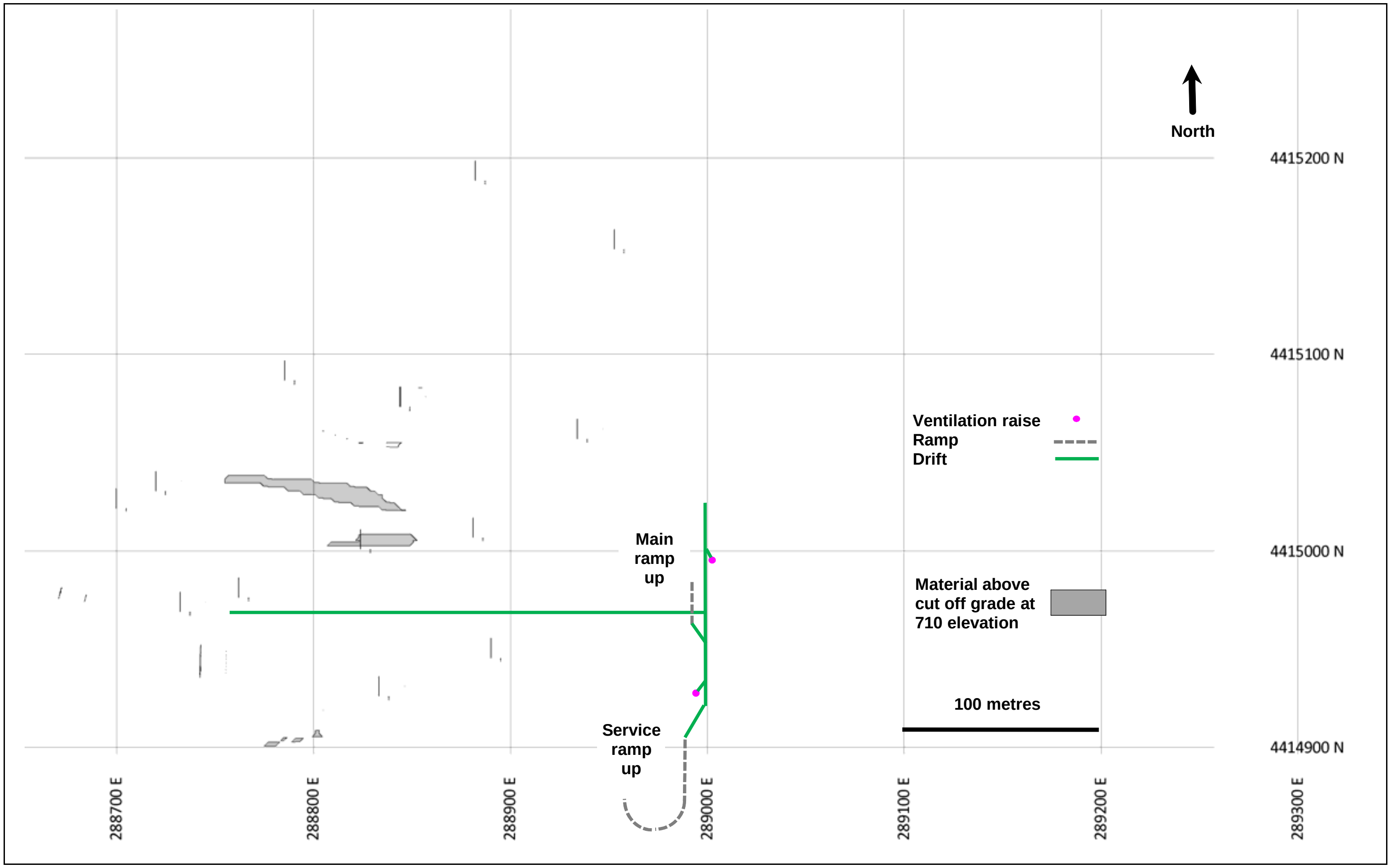


Figure 19.5 Typical Ore Definition Drilling on 1010 Level (plan)

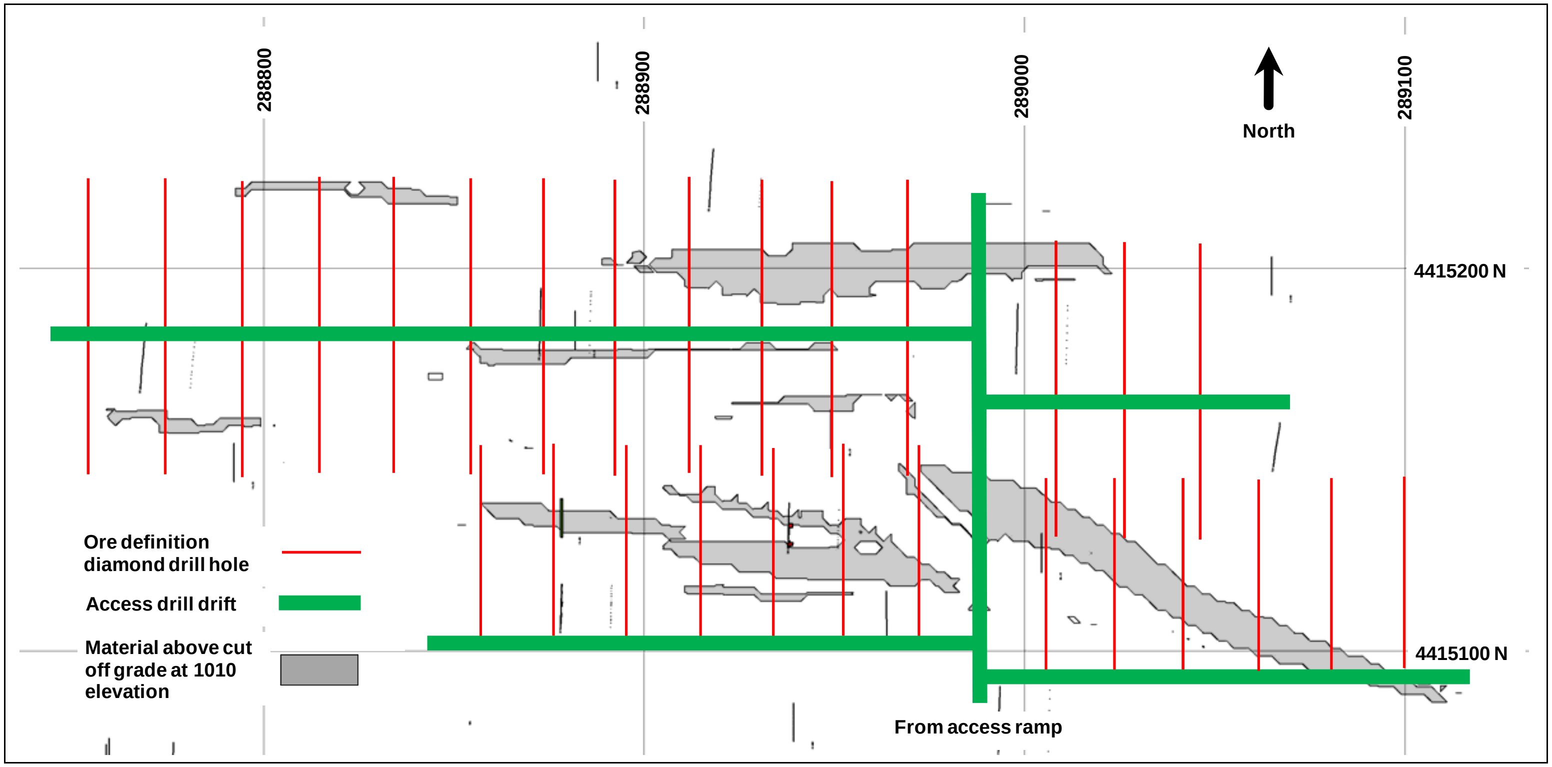
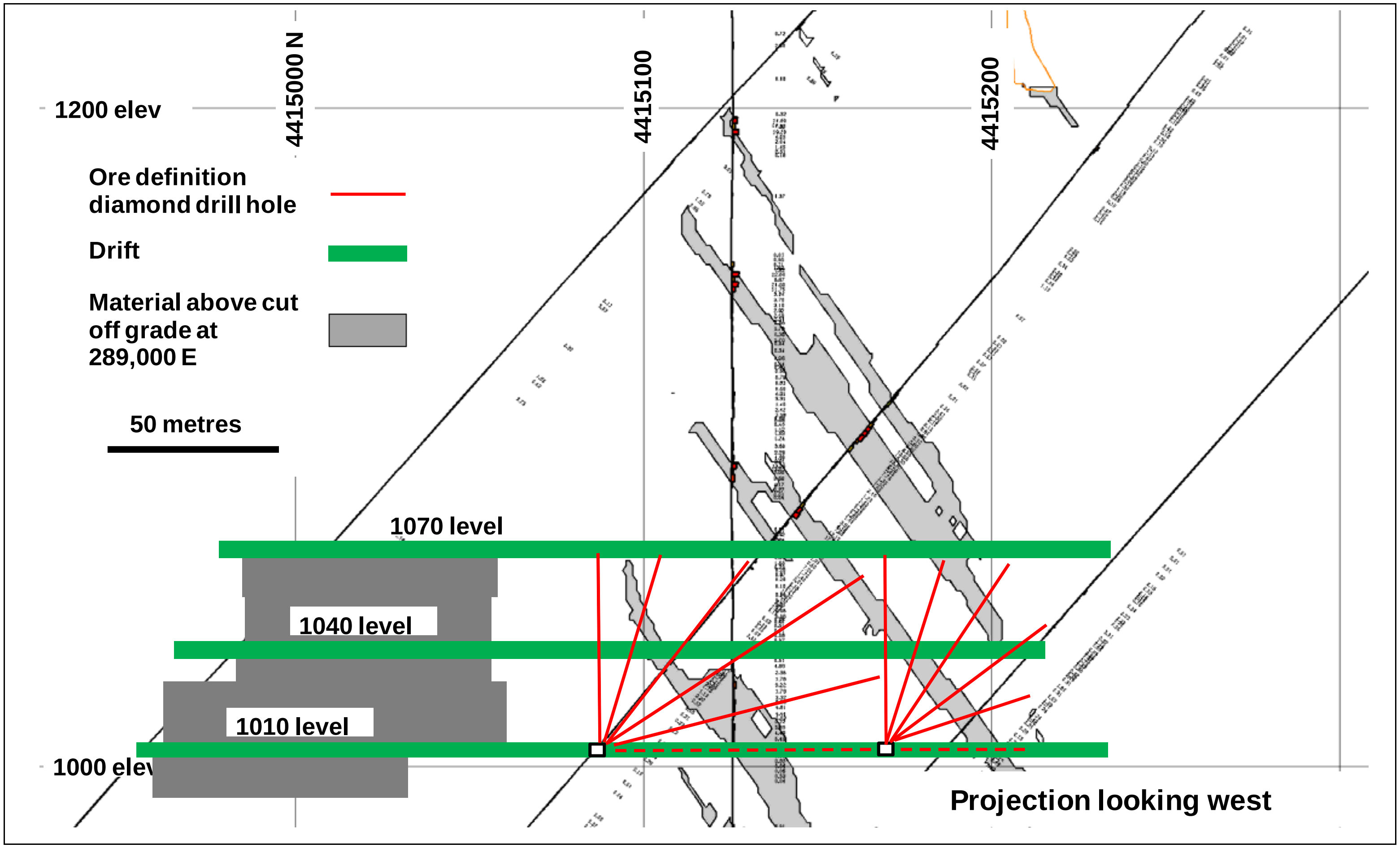


Figure 19.6 Typical Ore Definition Drilling on 1010 Level (section)



19.1.5.4 Mining Method

The zones of above cut off grade material appear to be reasonably continuous, steeply dipping and with no major changes in dip or strike resulting from folding, faulting or other geologic features. This geometry suggests the suitability of longhole mining. To improve ground stability and maximize extraction of the resource the stopes will be filled with uncemented rock fill.

The maximum horizontal width of the zones appears to be about 15 m and it is assumed that longitudinal stopes will be viable and stable. They will have a maximum length of 70 m with 10 m strike length pillars between stopes.

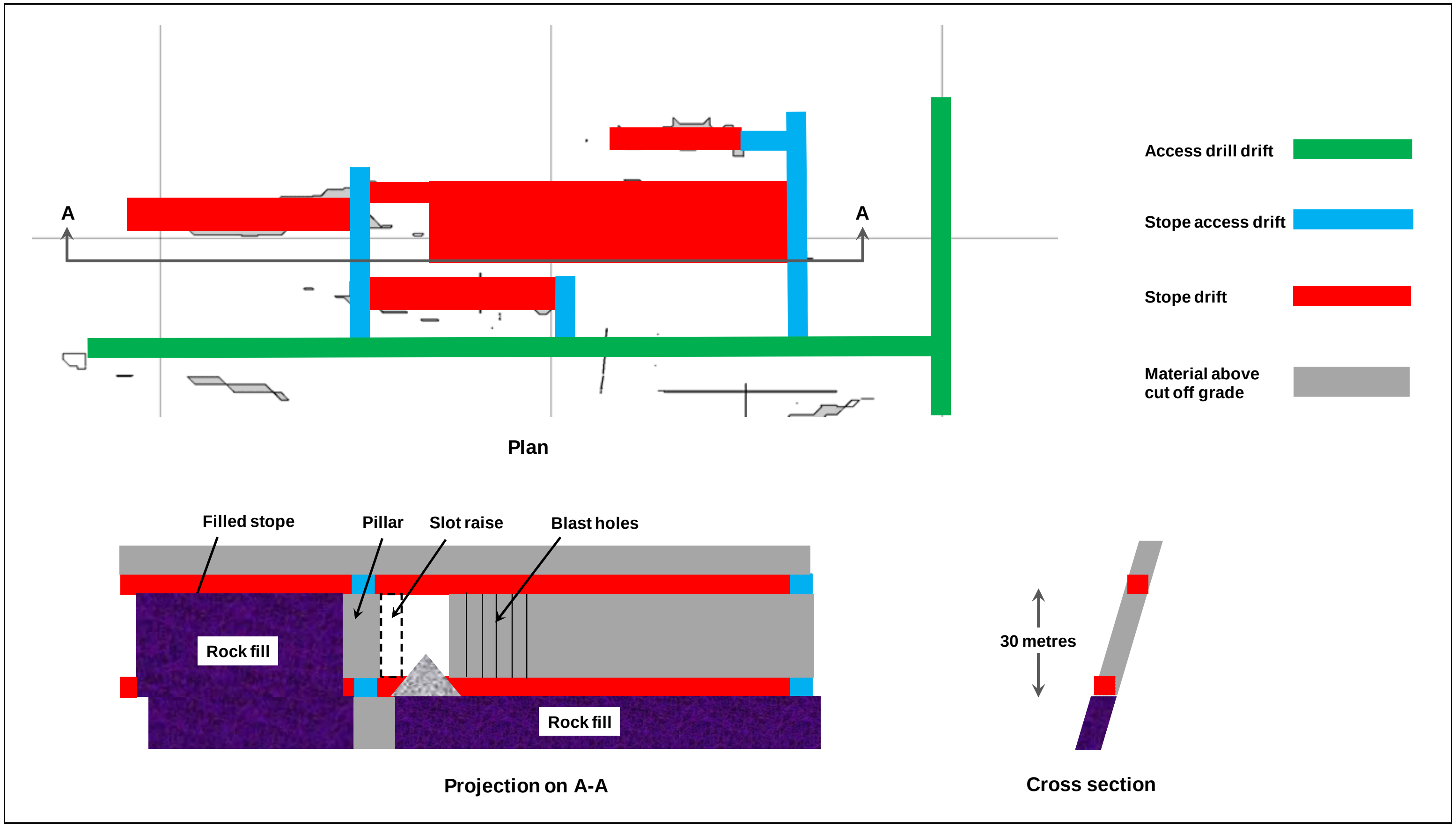
If the zones are discovered to be significantly wider then an alternative is to use transverse longhole stoping with the primaries filled with cemented fill. This would increase unit mining costs. The overall mining extraction, recovery and dilution would be relatively unchanged. Current interpretations of the deposit geology do not appear to indicate the need for this type of stoping.

Because of the vertical extent of the deposit it is proposed to establish three production horizons. The first will be on the 1010 m level, followed by the 830 m level and then the 710 m level. Stoping will advance up dip from each horizon. The mineable zones appear to have variable outlines from level to level and generally to have limited vertical extent. With appropriate design of the stopes this will obviate the need to leave major sill pillars or to place cemented fill at the 1010 and 830 production horizons.

Fill is necessary to maintain overall ground stability when mining multiple sub parallel zones. To improve stability rock fill can be dumped at the opposite end of the stope from the active face concurrent with production to limit the extent of the open void. Also cable bolts can be installed in the walls prior to stoping.

Two active production horizons and the strike length of the mineable material of up to about 600 m will provide multiple mining areas. Even with the strict stope sequencing required to ensure overall ground stability when mining sub parallel zones it is considered that the combination of production horizons and strike lengths will allow the planned mining rate of 3,500 tpd to be achieved.

Figure 19.7 Longitudinal Retreat Stoping with Rock Fill



19.1.5.5 Stope Development

From the initial drill drifts lateral access will be developed to intersect the mineable zones (Figure 19.7). A drift will be developed along the strike of each zone. From face and wall mapping and sampling, and possibly additional short core hole drilling, the economic stoping limits will be determined.

When the complete stoping length is developed a slot raise will be developed at one end between the levels. The lengths of individual stopes will be limited to 70 m and in longer zones 10 m pillars will be left between the stopes and additional slot raises developed. The slots will be developed as drop raises and expanded to the full stope width using rings of parallel blast holes.

The first stope in each zone will be developed to the full zone width by combinations of drifting and slashing. Rings of blastholes will be drilled down from the upper level and blasted retreating from the slot to the access drift. Blastholes will be drilled to the full stope width on the upper level with additional short drill drifts and trim rings when required.

Mucking of broken ore will be by remote controlled LHD's on the lower level with the ore hauled to a truck loading area in the access drift.

Stopes will be filled with rock fill from surface quarrying and underground development waste rock. Waste rock from surface quarrying will be sized on surface and dumped into the fill transfer raise which will be equipped with control chains and chutes on the main levels.

Development waste rock will be used as fill when practical and will be dumped into the fill raise or hauled directly from the heading to a stope. This will provide a relatively small amount of the total fill placed.

Trucks will dump rock fill from the upper level into the stope on the completion of ore mucking. In the longer zones and if ground stability is of concern then filling can proceed concurrently with mucking by dumping from the opposite end of the stope. This technique, known as Avoca stoping, can be used to limit the open length to a stable span.

19.1.5.6 Mobile Mining Equipment

Lateral development will be by two boom electric hydraulic drill jumbos, with mucking by diesel LHD's and bolting for ground support by electric hydraulic bolting jumbos. Mucking will be to a remuck cut out for later loading by LHD into diesel trucks for haulage to a disposal point. During development of the decline from surface to the 1010 m level all of the waste rock will be truck hauled to surface. For development below that level most waste rock will be hauled to a rock pass dump for use as stope fill.

In-the-hole (ITH) electric hydraulic production drills will be used to develop stope slot raises between the upper and lower ore drifts and to drill rings of production blast holes. Following blasting stopes will be "non entry" for personnel and no ground support will be installed.

Stope mucking will be by LHD, using remote control when necessary. Ore will be loaded into trucks in the access drifts and hauled to surface on the main ramp.

Trucks will be used to transfer material from the fill raise chutes to stopes. Some of these will be equipped with ejector boxes to facilitate dumping in areas with restricted headroom.

Other mobile equipment will be units typically used in the provision of mine services, including explosives delivery and loading vehicles, road graders, scissor lifts, utility vehicles, and personnel carriers.

Mobile core (diamond) drills will be used for mineralization outline drilling from the footwall ramps and level access drill drifts.

The major mobile equipment requirements for a typical year of full development and production are shown in Table 19.4.

Table 19.4 Major Mobile Mining Equipment

Item	Typical Model / Size	Number
Development load-haul-dump machine	5.0 m^3, diesel	5
Rock haulage truck	20.0 m^3, 50 t, diesel	8
Production load-haul-dump machine	7.0 m^3, diesel	4
Development drill jumbo	Electric hydraulic, double boom	5
Production drill	Electric hydraulic, ITH, 110 mm diameter	2
Bolting drill jumbo	Electric hydraulic, single boom	3
Blockhole drill	Electric hydraulic, single boom	1
Personnel carrier	22 person	2
Utility vehicle	Flatdeck, crane, explosives, fuel / lube	4
Scissor lift	For construction services	2
Maintenance vehicle	Mechanical, electrical	2
Personnel vehicle	8 person	4
Maintenance personnel vehicle	4 person	2
Utility vehicle	For mine services	8
Core drill	NQ capacity diamond drill	1
Small scissor lift	For technical services	2
Road grader	Low profile	2

19.1.5.7 Mining Equipment Maintenance

The main facilities for maintenance of mobile equipment will be on surface as the ramp system will allow access to and from all areas of the mine.

The underground shop facilities will be limited to those necessary to provide preventative maintenance services. The cost estimate includes allowances for the excavation and equipping of a shop in the central area of the mine.

19.1.5.8 Dewatering

There is no basis for estimating groundwater inflow into the mine. The cost estimates include allowances for an underground drainage system, collection sumps and a pumping system discharging to surface.

There are no allowances in the cost estimates for cover drilling or grouting during mining operations.

19.1.5.9 Ventilation

Ventilation airflow requirements are based on the forecast use of diesel equipment using typical legislated standards. The maximum airflow is forecast to be about 380 m^3/s. To provide this one primary intake and one primary exhaust raise will be developed and their ventilating capacities supplemented by the two ramp systems. Each raise will be equipped with fans at their surface collars to provide a push/pull system. The intake fans will be equipped with heating plants designed to raise the intake air temperature during winter. Assuming that there will be some groundwater inflow underground the cost estimate assumes heating the air to a few degrees above zero centigrade.

Connections will be made between the cross cut and the ventilation raises on each level and equipped with airflow controls. Ventilation of active mining areas will be provided by secondary fans and ducting.

19.1.5.10 Electric Power

Electric power will be supplied underground, typically at 4,160 V, using a ring system through the two ramp systems. Power centres with step down transformers will be located to provide power to drills, fans and face pumps in active mining areas and to fixed facilities including the pump room and the maintenance shop.

The annual power consumption during a typical year of full mining activity is estimated to be about 54,000 MWh with the ventilation fans and dewatering pumps being the major consumers.

19.1.5.11 Mining Supplies

Operating and maintenance supplies will be transferred underground by service vehicles using the service ramp from surface. Facilities for short term storage will be established underground.

Diesel fuel will be transferred via a cased borehole from surface to fuelling stations on the main levels.

The main explosive is assumed to be bulk ammonium nitrate/fuel oil (ANFO) with minor usage of packaged emulsion explosives. The explosives will be stored in surface magazines and delivered to underground magazines for short-term storage. Mobile transport and loading equipment will transfer explosives to the mining areas.

19.1.5.12 Mining Personnel

The number of operating and maintenance personnel and those required to provide management, supervisory and technical services is forecast to vary between 135 to 160 on site each day depending on the stage of the mine life and level of activities. Cost estimates assume mining two shifts per day for seven days per week with personnel working four days on and three days off.

Office and change house facilities will be located in the surface building complex at the plant site.

Portable refuge stations will be located in active underground mining areas and will also serve as lunchrooms and be equipped with toilets. An emergency personnel warning system will be installed, egress routes marked and personnel trained in mine rescue duties.

19.1.6 Schedules of Mine Development and Production

Portal construction for the two declines is scheduled to start at the same time. The 1130 m level is reached in the third quarter of project year two. This will provide sites for setting up equipment to bore the upper sections of the service raises. Rock handling, ventilation and dewatering facilities will be installed. The 1010 m level is reached in the first quarter of year three and stope development on that level is scheduled to start in the second half of the year. Stope development starts on the second production horizon, on 830 m level, in year five.

Ore production in the fourth quarter of year three is forecast to average about 1,000 t/d. As more stopes are developed the production rate is forecast to increase to the design rate of 3,500 t/d by the middle of year four. This rate is sustained until the end of year twelve and then decreases until mining is completed late in year thirteen.

The forecasts of annual underground mining activities and of mine production are shown in Tables 19.5 and 19.6 respectively.

Table 19.5 Annual Mine Activities

	Units	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Total
Development															
Ramp	m	2,160	2,590	1,640	1,300	1,300	1,300	100	-	-	-	-	-	-	10,390
Access drift	m	-	-	1,910	2,590	2,330	1,300	2,500	2,350	1,710	780	600	-	-	16,070
Stope drift	m	-	-	920	2,640	1,770	1,130	1,680	1,880	2,640	1,520	2,040	-	-	16,220
Total lateral		2,160	2,590	4,470	6,530	5,400	3,730	4,280	4,230	4,350	2,300	2,640	-	-	42,680
Drop raise	m	-	-	440	1,170	810	550	830	750	1,330	520	700	-	-	7,100
Bored raise	m	-	-	990	-	540	-	330	-	-	-	-	-	-	1,860
Total raise		-	-	1,430	1,170	1,350	550	1,160	750	1,330	520	700	-	-	8,960
Total development		**2,160**	**2,590**	**5,900**	**7,700**	**6,750**	**4,280**	**5,440**	**4,980**	**5,680**	**2,820**	**3,340**	**-**	**-**	**51,640**
Diamond drilling	m	-	-	50,100	67,900	61,100	34,100	65,500	61,600	44,800	20,500	15,700	-	-	421,300
Haulage															
Development waste	t	148,700	178,500	269,600	267,700	263,300	178,500	186,900	161,800	117,900	53,500	41,400	-	-	1,867,800
Stope and development ore	t	-	-	84,000	1,210,800	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	759,300	12,134,100
Total		**148,700**	**178,500**	**353,600**	**1,478,500**	**1,523,300**	**1,438,500**	**1,446,900**	**1,421,800**	**1,377,900**	**1,313,500**	**1,301,400**	**1,260,000**	**759,300**	**14,001,900**
Rockfill (1)	m^3	-	-	-	270,200	305,300	321,400	307,200	303,300	281,900	312,900	299,800	350,000	210,900	2,962,900
Production	t ore	-	-	84,000	1,210,800	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	759,300	12,134,100

Notes

(1) Includes the development waste rock used as fill

Table 19.6 Annual Mine Production

	Units	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Total
Material mined	t	-	-	84,000	1,210,800	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	759,300	12,134,100
Grade	%Zn	-	-	5.7	5.7	5.6	6.4	5.7	5.1	5.3	5.3	4.3	4.3	3.4	5.2
	%Cu	-	-	0.4	0.3	0.4	0.4	0.3	0.3	0.4	0.5	0.5	0.3	0.4	0.4
	%In	-	-	37.0	29.7	22.2	25.4	41.3	49.1	38.6	33.0	35.1	38.4	37.9	35.0

Table 19.7 Annual Operating Costs

	Units	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Total
Total	$1,000's	-	-	9,201	43,997	44,491	37,308	44,542	42,064	40,748	36,121	36,426	33,407	24,473	392,779
Average	$/t mined	-	-	109.53	36.34	35.31	29.61	35.35	33.38	32.34	28.67	28.91	26.51	32.23	32.37

19.1.7 Operating Costs

19.1.7.1 Basis of Estimate

Forecasts of annual mining activities, including development, outline diamond drilling, stoping, filling and rock haulage were prepared. Unit costs for these activities were estimated based on current prices of labour and materials and assuming typical productivities.

Allowances were included for general mine expenses covering the costs of personnel for supervision, technical services, repair and maintenance of equipment, and for other services not directly included in production, and for electric power and the fuel for mine air heating.

The unit costs were applied to the activity forecasts to estimate the annual underground mining costs. The operating costs for mobile mining equipment were included in the unit costs based on typical consumption of fuel, lubricants and usage of repair and maintenance items. The costs for labour were based on current rates for mines in the western USA and for supplies were based on recent budgetary quotations and in-house data.

19.1.7.2 Operating Costs

The annual mine operating costs and the overall life of mine cost per tonne of mineralized material mined are shown in Tables 19.7 and 19.8.

Table 19.8 Life of Mine Operating Costs

Activity	Cost ($/t)
Lateral development	4.07
Raise development	1.37
Development haulage	1.24
Diamond drilling	2.26
Stoping	4.12
Stope ore haulage	4.37
Filling	0.82
Fill haulage	0.41
Sample assaying	0.27
General mine expense	9.93
Electric power	3.51
Total	32.37

19.1.8 Capital Costs

19.1.8.1 Basis of Estimate

Preproduction development is assumed to be by contractor(s). Costs are estimated from unit rates based on current prices of labour and materials with allowances for contractors' typical overheads and profit and assuming typical productivities for ramp, drift and bored raise development. It was assumed that the owner purchases all mining equipment for the use of the contractor, except for temporary service items typically provided by a contractor during early mine development and included in the overhead cost.

The schedules for purchasing additional and replacement mobile mining equipment are based on typical rebuild periods for trucks and LHD's and typical useful lives for these and the drilling equipment. An annual allowance was included for the replacement of service equipment.

The costs of the main items of mining equipment were based on recent budgetary quotations from major equipment suppliers and for underground facilities were based on recent estimates for similar projects and include allowances for indirect costs.

19.1.8.2 Capital Costs

Estimates of the annual costs for equipment, facilities and preproduction development are summarized in Table 19.9. Note that these do not include contingency allowances.

Table 19.9 Preproduction Capital Costs

	Costs ($M)			
Item	**Year 1**	**Year 2**	**Year 3**	**Total**
Mobile equipment	12.9	7.3	18.6	38.8
Fixed facilities	1.4	5.1	6.1	12.6
Preproduction development	13.9	15.0	18.8	47.7
Total	28.2	27.4	43.5	99.1

The initial capital costs for underground mobile equipment are estimated at $38.8 M, for fixed facilities at $12.6 M, and for preproduction development at $47.7 M. Including sustaining expenditures the life-of-mine capital costs are $141.1 M and are summarized in Table 19.10.

Table 19.10 Life of Mine Capital Costs

	Costs ($M)				
Item	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**
Mobile equipment	12.9	7.3	18.6	0.9	1.2
Fixed facilities	1.4	5.1	6.1	1.1	1.0
Preproduction development	13.9	15.0	18.8	-	-
Total	28.2	27.4	43.5	2.0	2.2

Item	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**
Mobile equipment	8.6	5.1	10.2	5.3	6.0
Fixed facilities	0.6	0.8	0.5	0.4	0.2
Preproduction development	-	-	-	-	-
Total	9.2	5.9	10.7	5.7	6.2

Item	**Year 11**	**Year 12**	**Year 13**		**Total**
Mobile equipment	-	-	-		76.1
Fixed facilities	0.1	-	-		17.3
Preproduction development	-	-	-		47.7
Total	0.1	-	-		141.1

19.2 Tailings Disposal

19.2.1 Introduction

Knight Piésold (KPL) completed a desktop conceptual engineering study for the Crypto Project in Utah, USA. The study included preliminary layouts for a Tailings Storage Facility (TSF), surface waste rock dump, and associated water management diversions to support a Preliminary Economic Assessment (Galbraith, 2010). Their report is included as Appendix D.

19.2.2 Site Characteristics

The project site is located on the western side of a small mountain range on the south end of the salt flats at the south end of the Great Salt Desert. The site ranges in elevation from approximately 1,320 m along the flats to approximately 1,800 m. The deposit is located at an approximate elevation of 1,350 m. The project site is located in an arid environment with minimal precipitation and high evaporation.

19.2.3 TSF Design Concept

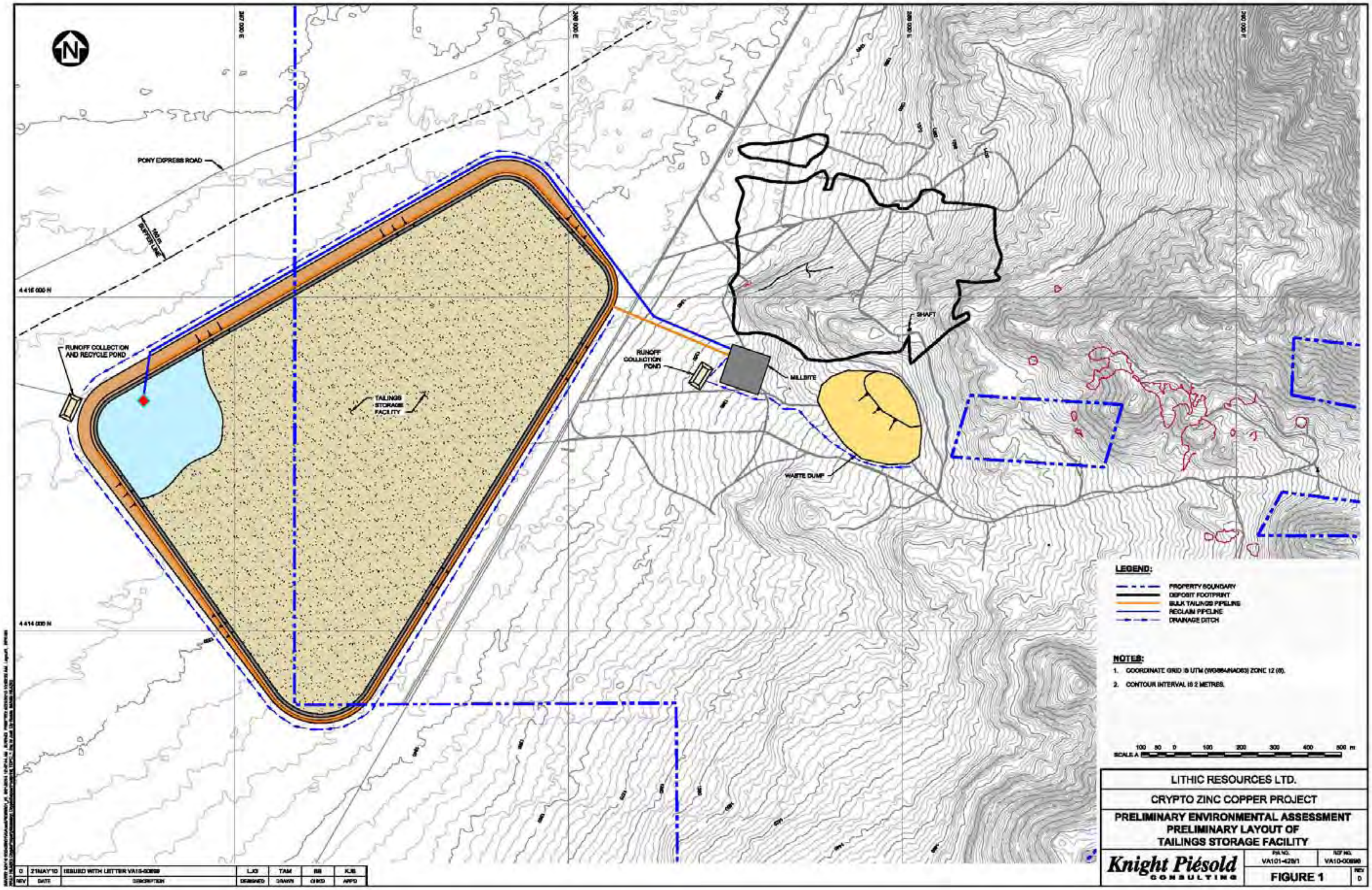
The objective of the TSF design concept was to locate a TSF as close a possible to the deposit while minimizing embankment fill requirements. It was also a requirement to maintain a 150 m buffer zone from the Pony Express Road. The TSF was thus located on the gentle sloping (1 to 2% grade) ground immediately west of the deposit between the Pony Express Road and the main road to the west of the deposit. The preliminary layout of the TSF included a large impoundment footprint, as the volumetric storage efficiency (volume of storage vs. volume of embankment fill materials) for TSFs on relatively flat terrain increases as the TSF footprint increases. The embankment fill materials (and costs) are comparatively lower than for facilities with smaller footprints and higher embankments.

The preliminary layout for the TSF was to provide containment for 12.1 million tonnes of tailings solids at an assumed dry density of 1.4 tonnes/m^3. The TSF has been located on the flats to the west of the deposit where the elevation ranges from approximately 1,324 m to 1336 m (Figure 19.8). The ultimate elevation of the TSF embankments is approximately 1,340 m and has embankments that range in height from 16 m at the western corner of the facility to approximately 4 m along the eastern side of the TSF.

The TSF covers an approximate area of 1.1 Mm^2 and includes a low permeability basin liner/subgrade material with a basin underdrain for seepage control. The embankments were assumed to be constructed using the downstream construction method with a 3H:1V upstream slope to facilitate placement and compaction of the low permeability basin liner, and a 2H:1V downstream slope. The embankments are homogeneous dams with appropriate filter zones to prevent internal erosion of the embankment fill materials in the event the liner system if damaged. The embankments were assumed to be constructed using local borrow materials.

The low permeability basin liner/subgrade material was assumed to be appropriate for seepage control at this stage. This should be revisited once the tailings geochemistry is better understood.

Figure 19.8 Schematic Site Plan with Proposed Tailings Storage Facility



The tailings were assumed to be conventional slurry tailings with a solids content of 30%, as per discussions with Lithic resources. The tailings were discharged from the embankment crest around the impoundment to maximize the tailings distribution within the TSF to improve the storage efficiency. A small supernatant pond was assumed to be located on the western side of the facility for the PEA.

The starter dam sized for containment of two years of operations at the full throughput. A mill ramp-up period was not considered for the PEA.

19.2.4 Water Management

The project site is located in an arid environment and there will be little water available to the process from local surface runoff. The water table is apparently relatively close to surface (approximately 1,300 m elevation) however, it is not known if a sustainable well field can be developed to provide the required water for the process, estimated at approximately 1,500 gpm for a throughput of 3,500 gpm at 30% solids. An allowance has been included in the cost estimate for make-up water which may involve sourcing water from across the valley to the west.

The make-up water requirements for the mill can be significantly reduced by thickening the tailings at the mill and capturing the thickener overflow rather than discharging the water to the TSF where it evaporates. Thickening tailings is usually cost effective in arid environments where water is in short supply. There are other potential options for optimizing the pipeworks and deposition systems to accommodate thickened tailings and these should be considered as potential opportunities to consider in future studies.

The management of runoff from storm events was not considered as part of the PEA. The layout of the TSF included a freeboard allowance of 2 m at all times, however, the adequacy of this assumption will be reviewed in future studies once the site hydrometeorology is better understood. The ultimate footprint of the TSF has embankments around the entire facility, which reduces the runoff from storm events to that generated from direct precipitation on the TSF. Prior to this, diversion ditches will be constructed to divert runoff from the upstream catchment around the TSF.

19.2.5 Closure and Reclamation

The TSF will be required to maintain long-term physical and geochemical stability, protect the downstream environment, and manage surface water. The preliminary closure and reclamation plan for the TSF includes constructing a TSF cover at closure and construction of an emergency overflow spillway.

19.2.6 Cost Estimate

The total capital cost over the mine life is approximately $47 million, with a total operating cost of approximately $4.6 million. The initial capital cost is approximately $19 million. This is a high level cost estimate and there may be opportunities to optimize the design concept once the site geotechnical conditions and tailings geochemistry are better understood. Additionally, the use of thickened tailings should be evaluated in future studies as this is typically cost effective in arid climates.

19.3 Economic Analysis

19.3.1 Introduction

The economic evaluation of the Crypto Project has been conducted using a simple discounted cash flow model and is based upon the following information:

- A mine production schedule provided by Keith Durston, P.Eng.
- A process flowsheet and metallurgical parameters provided by Ken Major, P.Eng.
- Capital and operating costs provided by Ken Major, P.Eng., Keith Durston, P.Eng. and Knight Piésold.
- Metal prices provided by Lithic Resources.

19.3.2 Cash Flow Model Inputs

19.3.2.1 Processed Mineral Resources and Mine Life

The mine plan has been developed for a mill throughput of 1,260,000 t/a. A total of 12.14 million tonnes with an average grade of 5.19% Zn, 0.38% Cu and 35 g/t In have been scheduled over a period of 11 years. The preproduction development and construction period is expected to be 2 years with an additional 2 years for exploration and Feasibility Studies. A preliminary comparison of mine development using shaft hoisting versus ramp access indicated that a better economic result was provided by the ramp development option hence that option has been the basis for the economic analysis.

19.3.2.2 Metallurgical Balance

The mine plan is based upon processing sulfides. Preliminary testwork completed by G&T of Kamloops, British Columbia was used to develop metallurgical projections for the sulfide resources. The recovery relationships for copper to copper concentrate and zinc and indium to zinc concentrate were used to assign net smelter returns (NSR) to each block in the resource model. Dilution for the NSR calculation was assumed to be 10% grade loss with volume losses equal to dilution gain. Grades were then reported on an undiluted basis for breakeven NSR levels for the purposes of mine planning where losses and dilution were reset for the purposes of scheduling based upon mining method. The production schedule and expected metallurgical recoveries are shown in Table Cash Flow Model.

Over the life of the mine, average recoveries are estimated to be 83.6% for zinc to zinc concentrate, 58.2% for Indium to zinc concentrate, 80.0% for copper to copper concentrate. Gold and silver are present in the copper concentrate. Based upon G&T testwork, these elements have been included in the cash flow analysis at concentrate grades of 21 g/t Au and 3,000 g/t Ag. Copper concentrate grade was estimated to be 32.0% Cu. Zinc concentrate grade was estimated to be 52.3% Zn.

19.3.2.3 Metal Prices

Metal price assumptions for the financial model base case are summarized in Table 19.11:

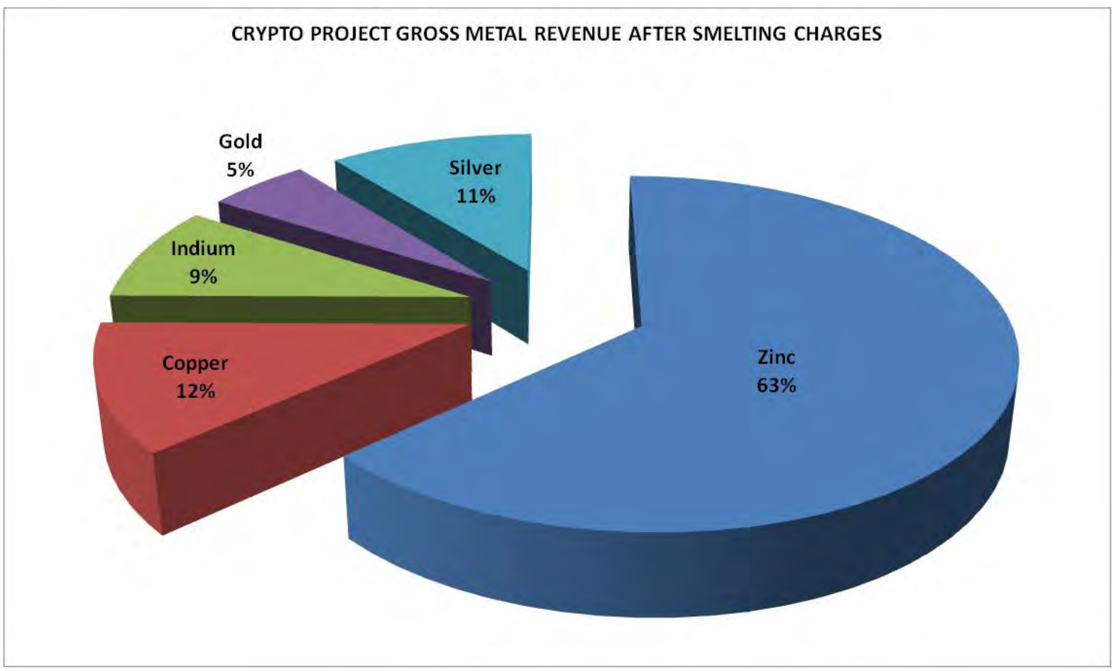
Table 19.11 Metal Price Assumptions

Metal Prices		US $
Zinc	Per lb	$1.10
Copper	Per lb	$2.00
Indium	Per kg	$500.00
Gold	Per oz	$850.00
Silver	Per oz	$12.00

In the early stages of the study, it became clear that a minimum processing rate of 3,500 tpd was necessary to maximize economies of scale. In order to support a minimum ten year mine life, the model considered all sulfide resources and used a zinc price of $1.10 per pound to outline sufficient tonnage above a cutoff grade of 4% zinc equivalent. The prices for other metals were the same as those used in MDA's resource estimate (Tietz et al, 2010)

The gross metal value distribution after smelting charges for the above price assumptions is shown in Figure 19.9.

Figure 19.9 Metal Value Distributions



19.3.2.4 Smelter Terms

Preliminary estimates for smelter terms for copper concentrate were based upon treatment at a local smelting facility. Treatment charges were assumed to be $50.00/dmt concentrate and refining charges were assumed to be $0.050/lb based upon recent smelter terms. Deductions were assumed to 3.4% for copper, 0.040 ounce/dmt gold and 1.0 ounce/dmt for silver. No penalties were expected.

Preliminary estimates for smelter terms for zinc concentrate were based upon treatment at a distant smelting facility. Base zinc treatment charges were expected to be $250/dmt with a base price of $2,000/t zinc based upon recent smelter terms. Unit deductions were assumed to be 8%. No penalties were expected.

Indium was also included in the revenue assumption with an estimated refining charge of $25/kg indium. The assumption can only be validated by direct negotiation with specific smelters.

19.3.2.5 Concentrate Shipping Costs

Transportation charges for trucking and rail transportation of zinc concentrate were estimated to be $75.00/wmt. Transportation charges for trucking of copper concentrate were estimated to be $22.50/dmt.

19.3.2.6 Royalties

Royalties payable to Vaaldiam have been estimated at 1.50% of gross revenue after smelter and concentrate transport charges. A one time payment of $1.0 million payable to Vaaldiam on obtaining development financing was included in the first year of production.

19.3.2.7 Capital Costs

Capital costs have been estimated for 4 years leading to production and for sustaining capital during operations. The first two years of development represent a sunk cost prior to construction of the process facility. The next two years are considered the development period leading to production.

Preproduction sunk costs including G&A, feasibility study, permitting and exploration were estimated to be $29.7 million. Preproduction capital cost was estimated to be $186.9 million and sustaining capital was estimated to be $118.9 million. An allowance was made for $20.0 million in working capital in Year 1 of operations.

19.3.2.8 Operating Costs

Operating costs have been estimated for mining, processing, tailings water management and general administration. These are summarized on a unit cost basis in Table 19.12.

Table 19.12 Operating Cost Summary

Operating Costs	US$/t ore
Mining	32.36
Processing	21.66
Tailings Water Management	0.25
G&A	10.00
Total Operating Costs	**$64.28**

19.3.3 Cash Flow Model Results

The cash flow model is based upon 100% equity funding and is included in Appendix E. Net present value (NPV) for the project has been estimated using discounted cash flow methods for a range of discount rates as summarized in Table 19.13.

Table 19.13 Pre-Tax NPV at Various Discount Rates

Discount Rate	NPV (US$)
2%	$7,752,289
4%	($22,694,300)
5%	($35,397,036)
6%	($46,668,188)
8%	($65,531,419)
10%	($80,340,706)
12%	($91,920,604)
15%	($104,612,224)

At a 10% discount rate, the project pre-tax NPV from beginning of the development period is -$80.3 million. The internal rate of return is 2.5%. The payback period is equal to the mine life.

Figures 19.10, 19.11 and 19.12 respectively show cumulative cash flow, mill head grades and zinc production costs net of credits. Table 19.14 shows annual production of payable metals.

Figure 19.10 Cumulative Cash flow

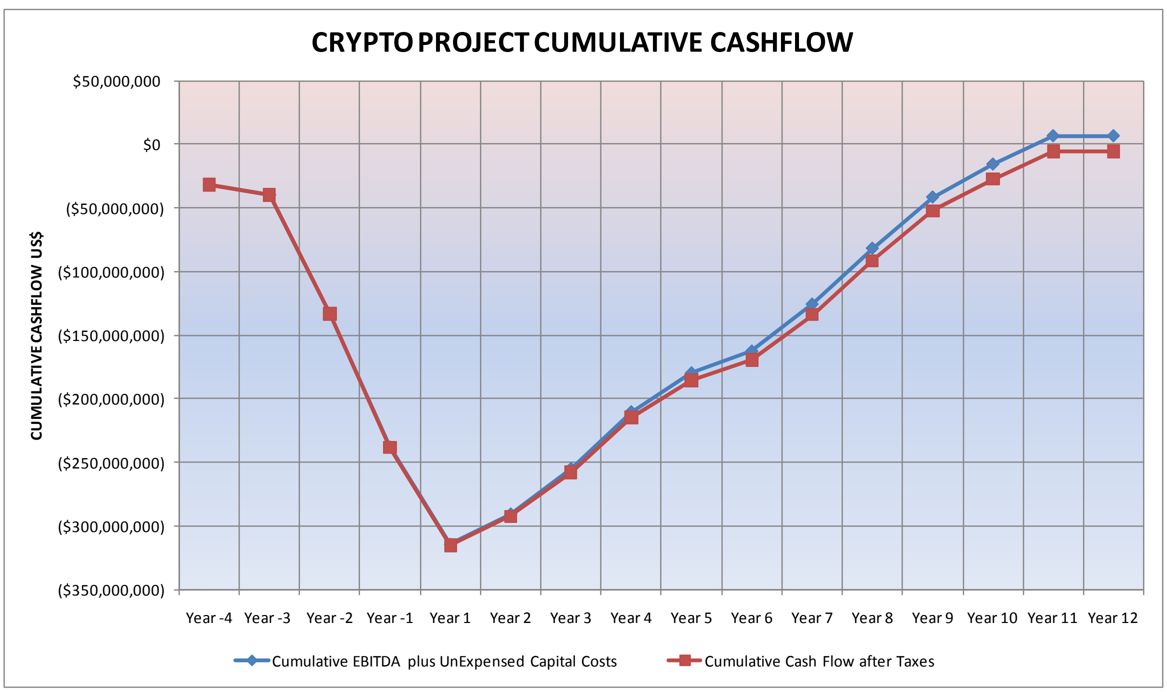


Figure 19.11 Mill Feed Head Grades

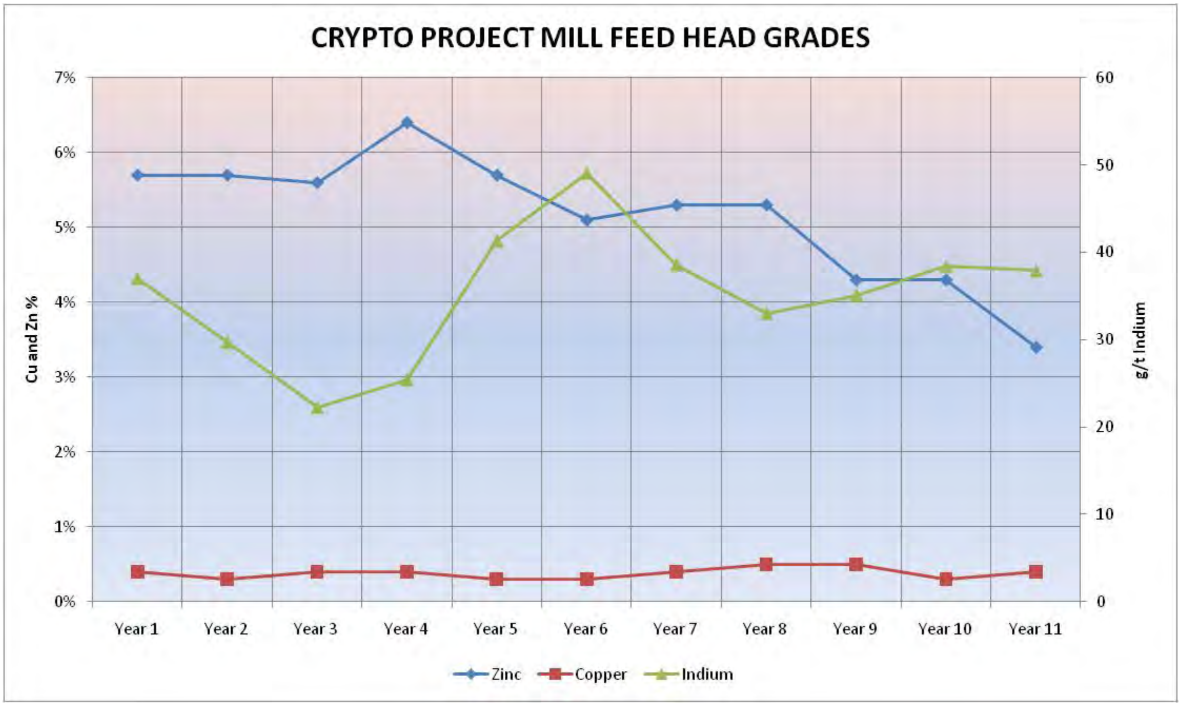


Figure 19.12 Zinc Costs Net of Co-products

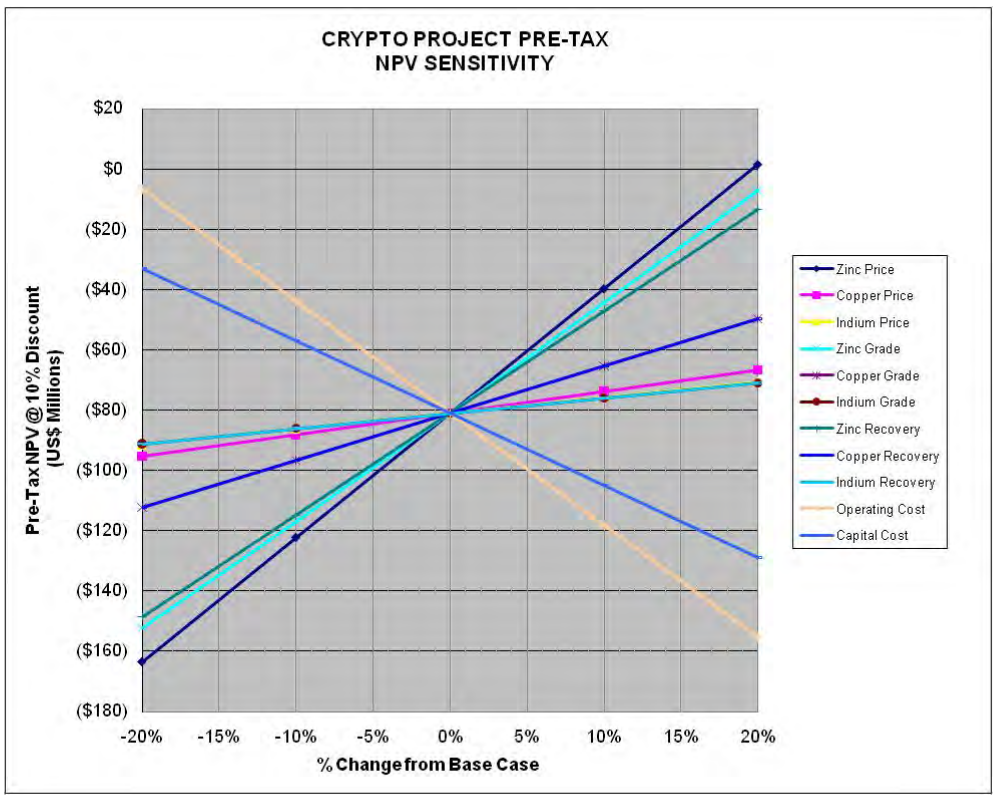
Table 19.14 Production of Payable Metals

	Lb Zinc	Lb Copper	Kg Indium	Oz Gold	Oz Silver
Year 1	7,534,132	572,744	1,686	566	80,997
Year 2	108,599,127	6,191,777	19,421	6,120	875,632
Year 3	110,825,078	8,591,167	14,891	8,491	1,214,951
Year 4	128,524,489	8,591,167	16,526	8,491	1,214,951
Year 5	113,011,975	6,443,376	28,228	6,368	911,213
Year 6	100,000,000	6,443,376	34,239	6,368	911,213
Year 7	104,308,149	8,591,167	26,739	8,491	1,214,951
Year 8	104,308,149	10,738,959	22,860	10,614	1,518,688
Year 9	83,059,163	10,738,959	25,125	10,614	1,518,688
Year 10	83,059,163	6,443,376	27,487	6,368	911,213
Year 11	38,904,308	5,177,201	16,824	5,117	732,152
Life of Mine Totals	**982,133,733**	**78,523,271**	**234,025**	**77,610**	**11,104,650**

19.3.4 Sensitivity Analysis

Figure 19.13 illustrates the results of a sensitivity analysis of key economic inputs to the Crypto cash flow model. The most sensitive parameters in terms of equivalent percent change are zinc price, zinc grade, zinc recovery, operating cost and capital cost.

Figure 19.13 Pre-tax NPV Sensitivity



A grade increase of 30% to 40% is required to demonstrate reasonable economic results for the current model of a projected mine life of ten years at a processing rate of 3,500 tpd. Increasing the cutoff grade of the existing resources to the point at which the average zinc grade is increased by 30% results in a loss of approximately 40% to 45% of resources available for mine scheduling. Further exploration is required to locate additional relatively higher grade resources that could increase the overall grade of the mineable resource and which might also be available for early mine scheduling. Existing lower grade resources could be used to extend mine life after return on investment has been achieved.

20.0 INTERPRETATION AND CONCLUSIONS

Historic production in the Fish Springs district came from high-grade lead-silver ores from oxidized, carbonate replacement deposits prior to 1953. Since then, a total of eight companies have held the Crypto property, four of which, including Lithic, have carried out drilling. The most significant mineralization discovered to date consists of sphalerite with minor chalcopyrite occurring in a series of concordant to discordant skarns and replacement bodies in carbonate rocks in proximity to a quartz monzonite intrusive complex. Two mineralized zones have been identified – the Main and Deep zones, separated by the Juab fault. The Main zone is oxidized to an average depth of about 250 meters.

The first NI 43-101-compliant resource for the Crypto zinc-copper-indium project was estimated by Mine Development Associates in 2009 (Tietz et al, 2010). Table 20.1 summarizes resource estimates made for sulfide and oxide mineralization:

Table 20.1 MDA Resource Estimate

Zone	Cutoff ZnEq %	Tonnes	ZnEq %	% Zn	Lb Zn	%Cu	Lb Cu	g/t In	Kg In
Sulfide									
Indicated	3.00	5,800,000	6.60	4.44	568,151,000	0.309	39,446,000	48.8	283,100
Inferred	3.00	13,805,000	6.83	4.84	1,472,057,000	0.372	113,191,000	37.4	516,400
Indicated	6.00	2,411,000	9.91	7.43	394,929,000	0.329	17,491,000	58.3	140,700
Inferred	6.00	6,297,000	9.91	7.62	1,057,338,000	0.443	61,497,000	41.7	262,300
Oxide									
Indicated	1.00	1,114,000	5.48	4.54	111,413,000	0.263	6,449,000	10.31	11,500
Inferred	1.00	4,644,000	4.45	3.73	382,312,000	0.165	16,932,000	12.55	58,300

**ZnEq (zinc equivalent) calculated at Zn=$0.80/lb, Cu=$2/lb, In=$500/kg; base case for sulfides at 3% ZnEq cutoff*

MDA's report stated that that there was good potential for expansion of these defined resources to the east, west and south and also very good potential for the discovery of new, similar zones beyond these extensions. Furthermore, it noted the very good possibility to increase resource classification just by getting more indium sample assays. The relatively small amount of Indicated material is due to the many fewer samples with indium grades because historic operators did not analyze for indium. It is very likely that much of the Inferred resource can be upgraded to higher classification. An upgrade in classification would also likely be accompanied by an as yet undetermined increase in indium grade, since the high-grade search restrictions used in the resource estimate constrain the high-grade and cause the Inferred material to be of generally lower grade than Indicated material.

MDA's report recommended a substantial program of followup drilling for general resource expansion but also that a Preliminary Economic Assessment (PEA) be carried out in order to determine whether continued drilling should emphasize resource expansion or focus on increasing the confidence level of the resource in anticipation of a pre-feasibility study.

A PEA was commissioned in early 2010 from a team of independent consulting engineers. Following a review of existing data, a conceptual mine plan was developed for underground extraction of the sulfide resource only as it was determined that mining of this portion of the deposit was most likely to have positive economics. While oxide mineralization was not included in the study, it represents a possible opportunity to be evaluated in future for open pit extraction following any eventual mining of the sulfide resource. Another potential opportunity not included in this study but to be evaluated in future is the possibility of generating a magnetite concentrate from the flotation tails.

Metallurgical recovery projections from preliminary testwork and a dilution factor of 10% were used to assign preliminary net smelter return (NSR) values to each block in the resource block model initially developed by MDA. Grades were then reported on an undiluted basis for breakeven NSR levels for the purposes of mine planning following which losses and dilution were re-applied on the basis of mining method and mine scheduling. Although not included in the original resource estimate, gold and silver were included in the PEA as substantial quantities of both elements were identified in the copper concentrate during metallurgical testwork.

Given the geological parameters of the deposit and likely development costs, a mining/processing rate of 3,500 tonnes per day was chosen to maximize economies of scale. In order to generate sufficient tonnage to support a ten year mine life, however, it was necessary to include all sulfide resources in the model, including lower grade material, and to use a zinc price of $1.10 per pound to outline enough tonnage above a cutoff grade of 4% zinc equivalent within that resource. Other metal prices were unchanged from MDA's resource estimate calculation with copper at $2/lb, indium at $500/kg, gold at $850/oz and silver at $12/oz.

Based on the geometry of the sulfide deposit, longitudinal retreat stoping with uncemented rock fill was chosen as the primary mining method for all mineable zones. A preliminary comparison of mine development using shaft hoisting versus ramp access indicated that a two-ramp development option was the superior choice. A total of 12.14 million tonnes with an average diluted grade of 5.19% Zn, 0.38% Cu and 35 g/t In were scheduled over a period of 11 years for a mill throughput of 1,260,000 t/a or 3,500 t/d.

A 3,500 tpd concentrator and a tailings facility along with offices and machine shops would be built at the site. A zinc concentrate with payable indium and a separate copper concentrate with payable gold and silver would be shipped to smelters. Personnel would be bussed in from surrounding communities. An existing substation and single phase power line servicing the property would be upgraded to accommodate the increased power requirements of the mining operation.

Cost and production parameters for this development scenario are summarized in Table 20.2. Assuming 100% equity funding, the pre-tax net present value (NPV) for the project at a discount rate of 10% is negative $80.3 million and the internal rate of return (IRR) is 2.5%. Analysis of the cash flow model indicates that the economics of the project are most sensitive, in order of importance, to zinc price, zinc grade, zinc recovery, operating cost and capital cost. Barring an increase in the price of zinc, an overall increase of 30% to 40% in zinc grade is required to achieve reasonable economic results for the mine development scenario contemplated in this study.

Table 20.2 Summary of Cost and Production Parameters

Pre-production capital	US$ millions	186.9
Sustaining capital	"	118.9
Total Capital	"	305.8
Mining/processing rate	tonnes per day	3,500
Mine life	years	11
Operating costs		
mining	US$/tonne	32.36
processing	"	21.66
tailings water management	"	0.25
G&A	"	10.00
Total operating cost	"	64.28
Average Annual Production		
zinc	lb	89,285,000
copper	lb	7,138,000
indium	kg	21,000
gold	oz	7,000
silver	oz	1,009,000

While its overall grade is too low to be economic at this point, the resource outlined to date does contain areas with higher grades. In aggregate, these represent approximately 55-60% of the minimum tonnage and grade required for an economic situation. Since there is good potential to expand existing resources to the east, west and south, and there is very good potential for the discovery of new zones beyond these extensions, it is entirely possible that enough additional mineralization at sufficient grade will be found to improve the potential economics of the project.

Further drilling is clearly warranted to explore for additional, relatively higher grade mineralization that could increase the overall grade of the mineable resource and which might also be available for early mine scheduling. Existing lower grade resources could be used to extend mine life after payback of development capital has been achieved.

21.0 RECOMMENDATIONS

21.1 Phase One Work Program

The next phase of work at Crypto should target expansion of higher grade resources through approximately 10,000 meters of core drilling with the following objectives:

- drill-testing open-ended extensions of existing resources with relatively large step-outs and focussing on areas with higher grades,
- the identification of new zones of mineralization away from existing resources but in similar stratigraphic settings, particularly in the region between the existing resources and skarn-type copper-zinc-molybdenum mineralization encountered in historic drill hole CC-43 located some 1,000 meters to the east, and
- update the resource estimate with new data and include gold and silver values.

An approximate budget (US$) for a Phase One program is given in Table 21.1:

Table 21.1 Phase One Exploration Budget

Task	**(US$)**
Exploration Drilling	2,170,000
Contractors and Other Personnel	600,000
Analytical	123,000
Resource Estimation	100,000
Camp and other Site Costs	283,000
Subtotal	**3,298,000**
~10% contingency	330,000
Budget Amount	**3,628,000**

21.2 Phase Two Work Program

Should the results of the Phase One program be positive, then this Preliminary Economic Assessment should be updated to determine the advisability of a Phase Two program which could include underground bulk sampling, continued metallurgical optimization, drilling, and advanced engineering studies as well as environmental work. The budget for all this work, which would form the basis of at least a pre-feasibility study, could be in the order of US$20 million.

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23.0 DATE AND SIGNATURE PAGE

Effective Date of report: September 17, 2010

The data on which the contained resource estimates are based was current as of the Effective Date.

Completion Date of report: September 17, 2010

"Ken Major"
Ken Major, P.Eng.
Date Signed: September 17, 2010

"John Nilsson"
John Nilsson, P.Eng.
Date Signed: September 17, 2010

"Keith Durston"
Keith Durston, P.Eng.
Date Signed: September 17, 2010

"Paul Tietz"
Paul Tietz, C. P. G.
Date Signed: September 17, 2010

"Steve Ristorcelli"
Steven Ristorcelli, C. P. G.
Date Signed: September 17, 2010

"Les Galbraith"
Les Galbraith., P.Eng.
Date Signed: September 17, 2010

"Giles Peatfield"
Giles R. Peatfield, Ph.D., P.Eng.
Date Signed: September 17, 2010

24.0 CERTIFICATE OF AUTHORS

I, Paul Tietz, C. P. G., do hereby certify that I am currently employed as Senior Geologist for Mine Development Associates, Inc. located at 210 South Rock Blvd., Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Biology/Geology from the University of Rochester in 1977, a Master of Science degree in Geology from the University of North Carolina, Chapel Hill in 1981, and a Master of Science degree in Geological Engineering from the University of Nevada, Reno in 2004.

2. I am a Certified Professional Geologist (#11004) with the American Institute of Professional Geologists.

3. 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. I am independent of the Issuer applying all of the tests in section 1.4 of National Instrument 43-101.

4. I am jointly responsible for sections 5-15, 17 and 18 of this technical report titled Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project, Juab County, Utah for Lithic Resources Ltd. dated September 17, 2010 ("Technical Report"). I visited the project March 26, 2008, and again June 9 through June 13, 2008.

5. I have not had prior involvement with the property that is the subject of this Technical Report.

6. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains the necessary technical information to make the technical report not misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated September 17, 2010.

"Paul Tietz"
Paul Tietz

Paul Tietz
Print Name of Qualified Person

AUTHORS' CERTIFICATES

I, Steven Ristorcelli, C. P. G., do hereby certify that I am currently employed as Principal Geologist by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Geology from Colorado State University in 1977 and a Master of Science degree in Geology from the University of New Mexico in 1980. I have worked as a geologist for a total of 32 years since my graduation from undergraduate university.

2. I am a Registered Professional Geologist in the states of California (#3964) and Wyoming (#153) and a Certified Professional Geologist (#10257) with the American Institute of Professional Geologists.

3. 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. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

4. I am jointly responsible for sections 5-15, 17 and 18 of this technical report titled Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project, Juab County, Utah for Lithic Resources Ltd. dated September 17, 2010 ("Technical Report").

5. I have not had prior involvement with the property that is the subject of the Technical Report. I visited the property on April 6, 2009.

6. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains the necessary technical information to make the technical report not misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated September 17, 2010.

"Steve Ristorcelli"

Steven Ristorcelli
Print Name of Qualified Person

AUTHORS' CERTIFICATES

I, Christiaan F. Staargaard, do hereby certify that I am President and Chief Executive Officer of Lithic Resources Ltd., 912-510 West Hastings St., Vancouver, B.C., Canada V6B 1L8 and:

1. I graduated with a Bachelor of Science degree in Geological Sciences from The Pennsylvania State University in 1977 and a Master of Science degree in Geochemistry from Queen's University in Kingston, Ontario in 1981. I have worked as a geologist in the field of mineral exploration since 1979 for a total of 30 years.

2. I am a Registered Professional Geoscientist (#20008) in the Province of British Columbia.

3. 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. However, I am not independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

4. My contributions to this technical report titled Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project, Juab County, Utah for Lithic Resources Ltd. dated September 17, 2010 ("Technical Report") have been under the supervision of Mssrs. Tietz, Ristorcelli, Major, Nilsson and Durston. I am not responsible for any sections of this report.

5. I had no involvement with the property that is the subject of the Technical Report before it was acquired by Lithic Resources in June 2005. I have visited the property on numerous occasions since then.

6. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains the necessary technical information to make the technical report not misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated September 17, 2010.

"C.F. Staargaard"

C.F. Staargaard
Print Name of Qualified Person

AUTHORS' CERTIFICATES

I, Kenneth W Major, do hereby certify that I am employed as a Mineral Processing Consultant for KWM Consulting Inc., 12576 – 206 St., Maple Ridge, BC, Canada, V2X 3M2.

1. I graduated with a Bachelor of Engineering, Metallurgical degree from McGill University in Montreal in 1976. I have worked as a metallurgist in the field of mineral processing for a total of 33 years since my graduation from university.

2. I am a Registered Professional Engineer in the Province of British Columbia (#13149).

3. 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. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

4. I am responsible for Section 16, Mineral Processing and Metallurgical Testing, and jointly responsible for Sections 20-21 of this technical report entitled "Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project, Juab County, Utah" for Lithic Resources Ltd. dated September 17, 2010 ("Technical Report").

5. I have not had prior involvement with the property that is the subject of the Technical Report.

6. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains the necessary technical information to make the technical report not misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated September 17, 2010.

"Ken Major "
Ken Major

Ken Major
Print Name of Qualified Person

AUTHORS' CERTIFICATES

I, Keith Durston, of 2204-153A Street, Surrey, British Columbia, V4A 4R5, do hereby certify that:

1. I graduated from the University of London, England, with a Bachelor of Science degree in Mining Engineering and Associateship of the Royal School of Mines in 1965. I have practiced my profession continuously since 1965 and have been involved in underground mining of gold, lead-zinc, copper-nickel, uranium and potash; open-pit mining of lead-zinc; reviews and evaluations of operating mines; and studies for the mining of gold, silver, lead-zinc, copper-nickel and diamonds from properties in the Americas, Europe, Africa, Asia and Australia

2. I am a Registered Professional Engineer in the Province of British Columbia (#8791).

3. 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. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

4. I am responsible for Section 19.1, Underground Mining, and jointly responsible for Sections 20-21 of this technical report entitled "Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project, Juab County, Utah" for Lithic Resources Ltd. dated September 17, 2010 ("Technical Report").

5. I have not had prior involvement with the property that is the subject of the Technical Report.

6. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains the necessary technical information to make the technical report not misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated September 17, 2010.

"Keith Durston"
Keith Durston

Keith Durston
Print Name of Qualified Person

AUTHORS' CERTIFICATES

I, John Nilsson, do hereby certify that I am employed as a Mining Engineering Consultant for Nilsson Mine Services, Ltd. of 20263 Mountain Place, Pitt Meadows, BC, Canada, V2X 3M2 and that:

1. I graduated with a Bachelor of Science degree from Queen's University in 1977 and a Master of Science degree in Engineering from Queen's University in 1990.

2. I am a Registered Professional Engineer in the Province of British Columbia (#20697).

3. 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. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

4. I am responsible for Section 19.3, Economic Analysis, and jointly responsible for Sections 20-21 of this technical report entitled "Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project, Juab County, Utah" for Lithic Resources Ltd. dated September 17, 2010 ("Technical Report").

5. I have not had prior involvement with the property that is the subject of the Technical Report.

6. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains the necessary technical information to make the technical report not misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated September 17, 2010.

"John Nilsson"
John Nilsson

John Nilsson
Print Name of Qualified Person

AUTHORS' CERTIFICATES

I, Giles R. Peatfield, P.Eng., do hereby certify that:

1. I am currently a self-employed Consulting Geologist with an office at 104-325 Howe Street, Vancouver, British Columbia, V6C 1Z7.
2. I am a Member of the Association of Professional Engineers and Geoscientists of British Columbia, and of the Geological Association of Canada, of the Canadian Institute of Mining and Metallurgy, of the Association of Applied Geochemists, and of the Society of Economic Geologists.
3. I have worked as a geologist for over forty years since graduation; as a graduate student, as an employee of a major mining company and for over 20 years as an independent consultant.
4. 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 fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am responsible for the preparation of the report included as Appendix B and am jointly responsible for the preparation of Section 14.3 of the technical report titled **Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project, Juab County, Utah** for Lithic Resources Ltd., dated September 17, 2010 (the "Technical Report") relating to the Crypto project. I have not visited the property.
6. I have not been intimately involved with the planning and execution of exploration on the subject property, but have reviewed the assay data, with regard to quality control issues, as well as making recommendations regarding assay procedures.
7. I have not had prior involvement with the property area that is the subject of the Technical Report, prior to Lithic Resources assuming control of the project.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101. For greater clarity, I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the property owner, nor do I to the best of my knowledge hold any securities in any corporate entity with property within a two (2) kilometre distance of any of the subject property.
10. I have read National Instrument 43-101 and Form 43-101F, and attest that the Technical Report has been prepared in compliance with that instrument and form.

Dated September 17, 2010.

"Giles R. Peatfield "
Giles R. Peatfield, Ph.D., P.Eng.

Giles R. Peatfield
Print Name of Qualified Person

AUTHORS' CERTIFICATES

I, Les Galbraith, do hereby certify that I am employed as an Engineer for Knight Piésold Ltd. of 1400 - 750 West Pender, Vancouver, B.C. Canada V6C 2T8 and that:

1. I graduated with a Civil Engineering Degree from from the University of British Columbia (1995).

2. I am a Registered Professional Engineer in the Province of British Columbia (#25493).

3. 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. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

4. I am responsible for Section 19.2, Tailings Disposal, of this technical report entitled "Preliminary Economic Assessment of the Crypto Zinc-Copper-Indium Project, Juab County, Utah" for Lithic Resources Ltd. dated September 17, 2010 ("Technical Report").

5. I have not had prior involvement with the property that is the subject of the Technical Report.

6. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains the necessary technical information to make the technical report not misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated September 17, 2010.

"Les Galbraith"
Les Galbraith

Les Galbraith
Print Name of Qualified Person

Appendix A

Crypto Project Mining Claims and Lease as of September 16, 2010

Crypto Project Mining Claims and Lease as of September 16, 2010

(From tabulation by Gatten, 2010)

Table A.1. Unpatented Mining Claims

Note: Crypto Zn 155 and Crypto Zn 159 are not visible on Figure 4.3 because they are overlapped by the Ogden, Last Chance, and Remnant patented claims.

Name	Number	Acres	Registered Owner	BLM Maintenance Due Date
Crypto Zn 150	359567	8.26	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 151	359568	13.09	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 154	359571	16.41	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 155	359572	4.13	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 156	359573	7.63	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 157	359574	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 158	359575	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 159	359576	5.74	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 160	359577	6.89	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 164	359581	12.40	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 165	359582	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 166	359583	12.40	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 167	359584	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 168	359585	12.40	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 169	359586	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 170	359587	6.20	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 171	359588	17.22	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 172	359589	13.77	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 173	359590	10.10	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 174	359591	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 175	359592	15.15	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 176	359593	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 177	359594	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 178	359595	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 186	359603	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 187	359604	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 188	359605	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 189	359606	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 190	359607	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 191	359608	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 192	359609	10.33	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 193	359610	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 194	359611	17.22	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 195	359612	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 196	359613	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 197	359614	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 198	359615	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 199	359616	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 200	359617	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
Crypto Zn 201	359618	20.66	N.P.R. (US), INC.	SEPT. 1, 2011

Name	Number	Acres	Registered Owner	BLM Maintenance Due Date
Crypto 1	378462	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 2	378463	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 3	378464	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 4	378465	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 5	378466	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 6	378467	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 7	378468	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 8	378469	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 9	378470	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 10	378471	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 11	378472	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 12	378473	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 13	378474	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 14	378475	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 15	378476	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 16	378477	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 17	378478	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 18	378479	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 19	378480	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 20	378481	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 21	378482	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 22	378483	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 23	378484	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 24	378485	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 25	378486	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 26	378487	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 27	378488	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 28	378489	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 29	378490	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 30	378491	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 31	378492	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 32	378493	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 33	378494	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 34	378495	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 35	378496	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 36	378497	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 37	378498	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 38	378499	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 39	378500	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 40	378501	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 41	378502	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 42	378503	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 43	378504	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 44	378505	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 45	378506	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 46	378507	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 47	378508	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 48	378509	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011

Name	Number	Acres	Registered Owner	BLM Maintenance Due Date
Crypto 49	378510	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 50	378511	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 51	378512	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 52	378513	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 53	378514	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 54	378515	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 55	378516	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 56	378517	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 57	378518	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 58	378519	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 59	378520	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 60	378521	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 61	378522	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 62	378523	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
Crypto 63	378524	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 9	404217	10.33	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 10	404218	10.33	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 11	386147	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 12	386148	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 13	386149	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 14	386150	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 15	386151	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 16	386152	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 21	404219	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 22	386158	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 23	386159	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 24	386160	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 25	386161	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 26	386162	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 27	386163	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 28	386164	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 29	386165	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 30	386166	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 31	386167	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 32	386168	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 33	386169	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 34	386170	20.66	LITHIC RESOURCES LTD	SEPT. 1, 2011
PONY 35	390306	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 36	390307	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 37	390308	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 38	390309	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 39	390310	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 40	390311	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 41	390312	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 42	390313	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 43	390314	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 44	390315	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 45	390316	20.66	N.P.R. (US), INC.	SEPT. 1, 2011

Name	Number	Acres	Registered Owner	BLM Maintenance Due Date
PONY 46	390317	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 47	390318	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 48	391816	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 49	390319	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 50	391817	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 51	391818	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 52	391819	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 53	391820	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 54	391821	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 55	390320	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 56	390321	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 57	390322	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 58	390323	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 59	390324	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 60	390325	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 61	390326	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 62	390327	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 63	390328	6.50	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 64	390329	5.00	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 100	404220	20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 101	404221	10.33	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 102	404222	5.15	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 103	404223	5.15	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 200		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 201		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 202		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 203		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 204		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 205		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 206		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 207		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 208		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 209		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 210		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 211		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 212		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 213		20.66	N.P.R. (US), INC.	SEPT. 1, 2011
PONY 214		20.66	N.P.R. (US), INC.	SEPT. 1, 2011

Table A.2 Patented Mining Claims

Name	Number	Acres	N.P.R. (US), Inc. % Ownership
Comstock	72037	17.421	100.000
Early Harvest	72037	20.081	100.000
Victor	72037	20.475	100.000
Last Chance	24027	17.06	100.000
Remnant	22719	10.63	100.000
Utah	22018	17.78	100.000
Niger	22191	19.62	100.000
Emma	22994	19.1	66.667
Nevada	24644	15.35	100.000
Mayflower	22109	20.26	83.333
Rubber	21966	6.33	100.000
Ogden	26188	16.25	41.667
Read Iron	22720	20.66	66.667
Galenia	22192	20.66	100.000
Meteor	33045	19.319	100.000
Bonny Doon	33045	17.959	100.000
Last Chance No. 2	n/a	7.35	12.500
Black Dragon	22850	19.69	100.000

Note: Acreage for claims not held 100% by NPR is the total claim acreage, not the adjusted acreage based on NPR's percentage of ownership.

Table A.3 Utah Mineral Lease

Lease Number	Acres	Ownership
ML 48312	611.3	N.P.R. (US), Inc.

Appendix B

Peatfield QA/QC Report

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MEMORANDUM

Date: **12 February 2009**

To: **C.F. Staargaard – Lithic Resources Ltd.**

From: **Giles Peatfield**

Re: **Summary of QC data – Lithic's Crypto project drilling – 2007, 2008:**

I have summarised the quality control ("QC") work for the core drilling campaign at Lithic's Crypto project in Utah, including work in 2007 and in 2008. This includes Lithic's drilling only; QC data for earlier drilling were not made available to me.

To complete this review, I have used all assay data generated by ALS-Chemex ("Chemex"), on samples submitted to their preparation laboratory in Elko, Nevada and analysed at their full service facility in North Vancouver. The assay data were sent to me directly by Chemex, and have not been passed through Lithic's hands.

I have considered several types of QC data, as detailed on following pages. The first is a set of standards monitoring data. I have included charts for Lithic's inserted standards for core drilling in 2007 and 2008. Certified standards were obtained from CDN Resource Laboratories Ltd. ("CDN") and from Geostats Pty. Ltd. ("Geostats"). The CDN standards were sulphide standards, with certified values for gold, silver, copper, lead and zinc; the Geostats standards were from supergene (oxidised) material and have certified values for, *inter alia*, silver, copper, lead and zinc. Also presented are charts to show the behaviour of the standards assays for cadmium, gallium, germanium and indium, although there are no certified values for these elements. The second data set involves assays for Lithic inserted "blanks". The third data set involves results of field, laboratory preparation and laboratory duplicate assays, where these are available.

Establishing the sampling and QC protocols for the Crypto project was an evolving process, as field personnel initially had little experience in rigorous procedures. There are still unresolved procedural issues that will be discussed in more depth in the conclusions section of this memorandum.

Note that there are two types of analyses to consider. The first, for lower metal contents, is the ME-MS61 ICP technique (herein "MS61"). The second, for greater concentrations, is the ME-OG62 technique (herein "OG62"). Gold analyses were by the Au-AA23 method (fire assay with an atomic absorption finish). Details of these analytical procedures can be seen on the ALS Laboratory Group website.

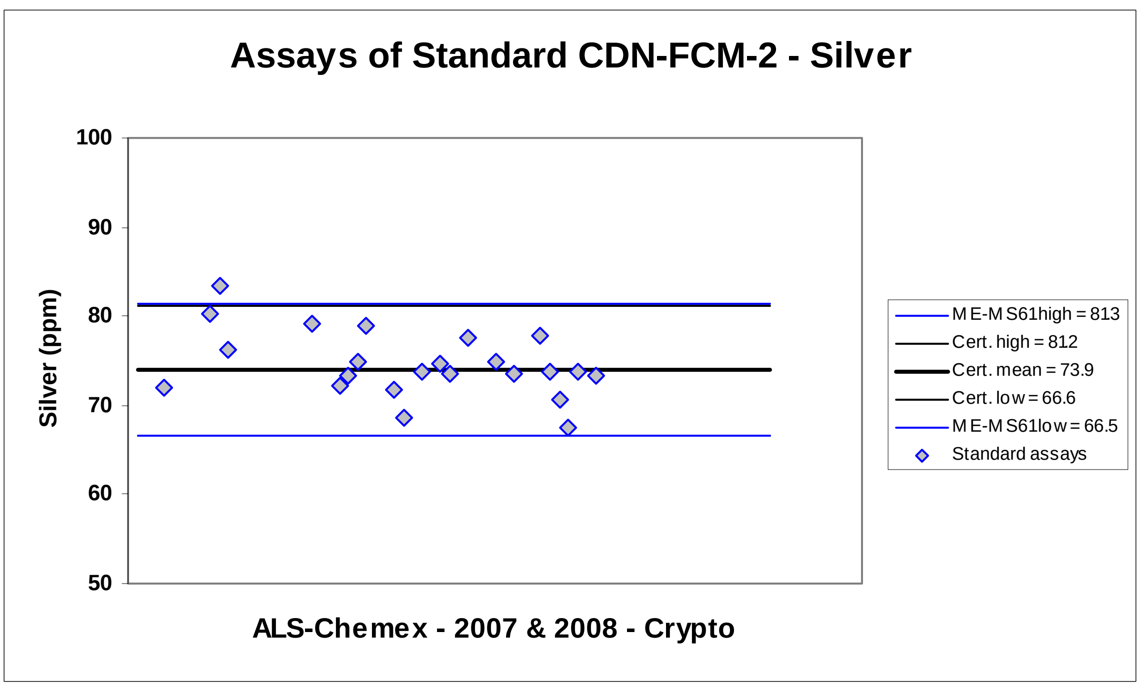
Standards monitoring plots – general comments:

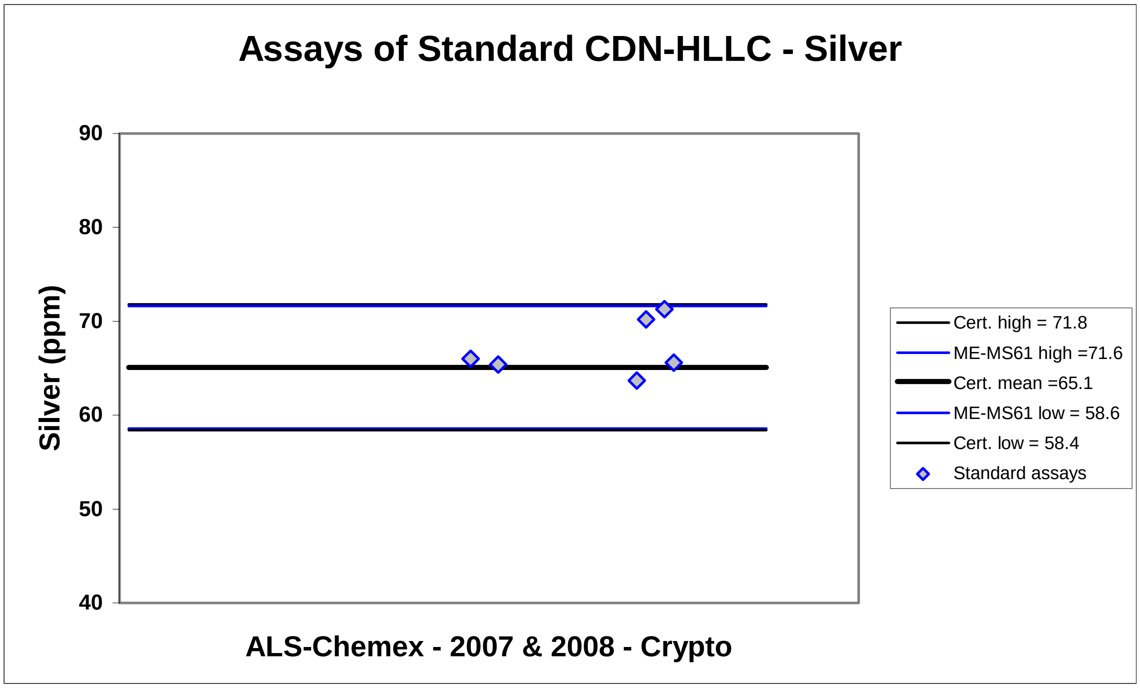
In the preparation of the monitoring charts for standards assays, I have used two sets of limits. The first is that given by the company that prepared and certified the standards. The second is a set designed for the analytical method, based on a formula provided by Chemex, which is in effect $\pm$ 7% of the mean with a small factor added. In the case of three of the standards used, which were obtained from CDN, there is very little difference between these sets of limits. In the case of the Geostats standards, the certified limits are much more restrictive than the limits calculated for the methods used.

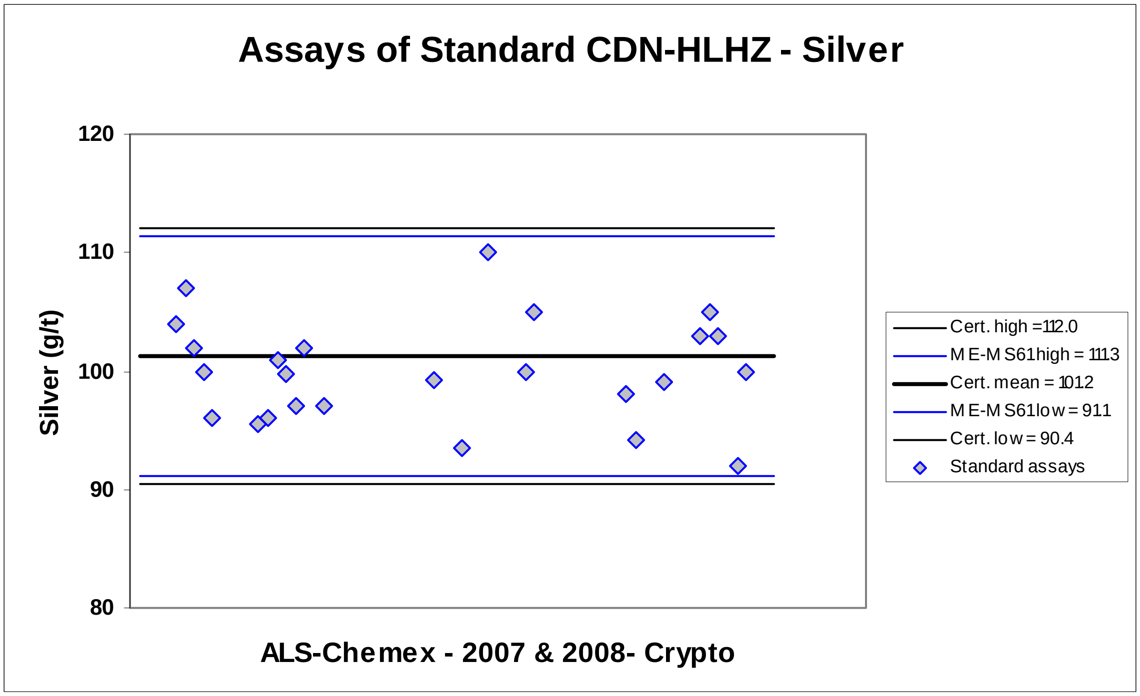
The CDN standards are: CDN-FCM-2 (high sulphide material from Campo Morado, Mexico); CDN-HLLC (high sulphide material from the High Lake deposit, Arctic Canada); and CDN-HLHZ (also high sulphide material from High Lake). The Geostats standards are: GBM396-10 (described as "Low Grade Supergene Pb/Zn ore ex Murchison") and GBM996-7 (described as "Gossan supergene sulphide ore ex NW QLD.")

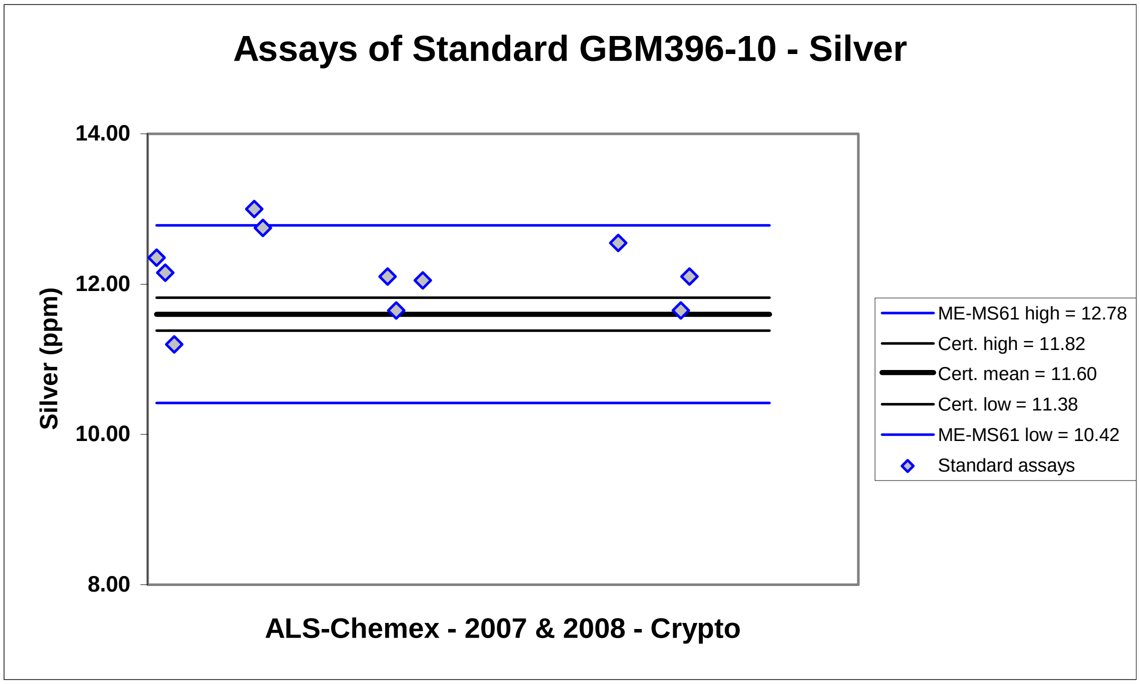
Monitoring plots for certified elements – Lithic inserted standards, 2007 & 2008:

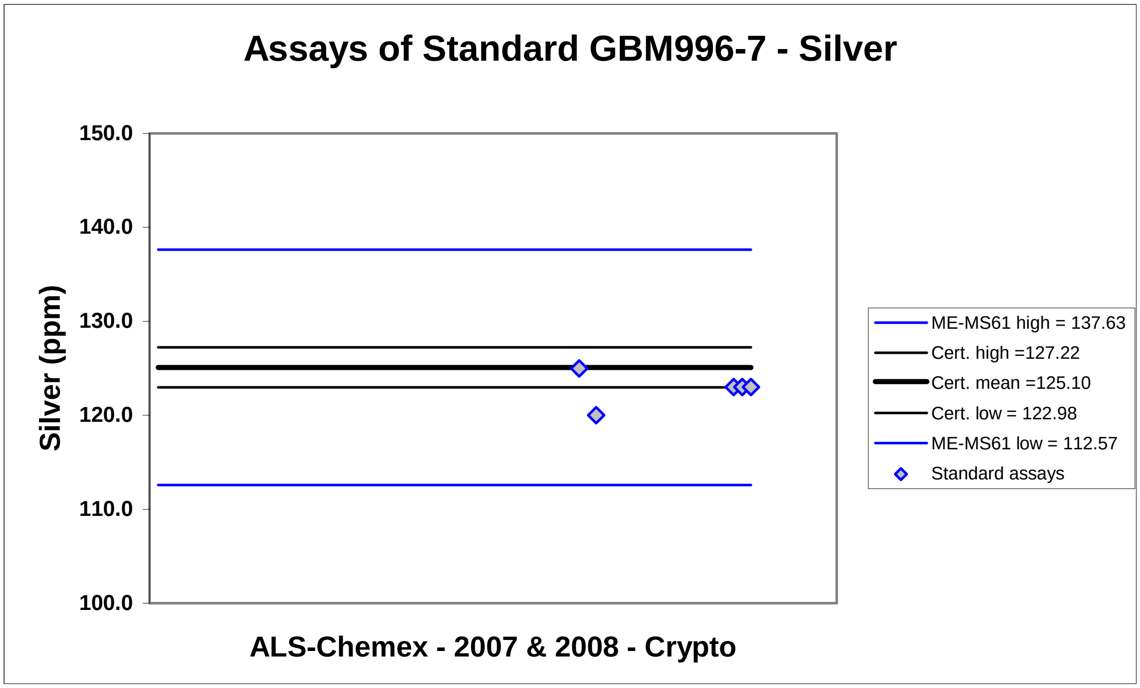
In 2007 and 2008, Lithic used three different CDN standards and two from Geostats, with values tracked for silver, copper, lead and zinc. Monitoring plots for these results follow:

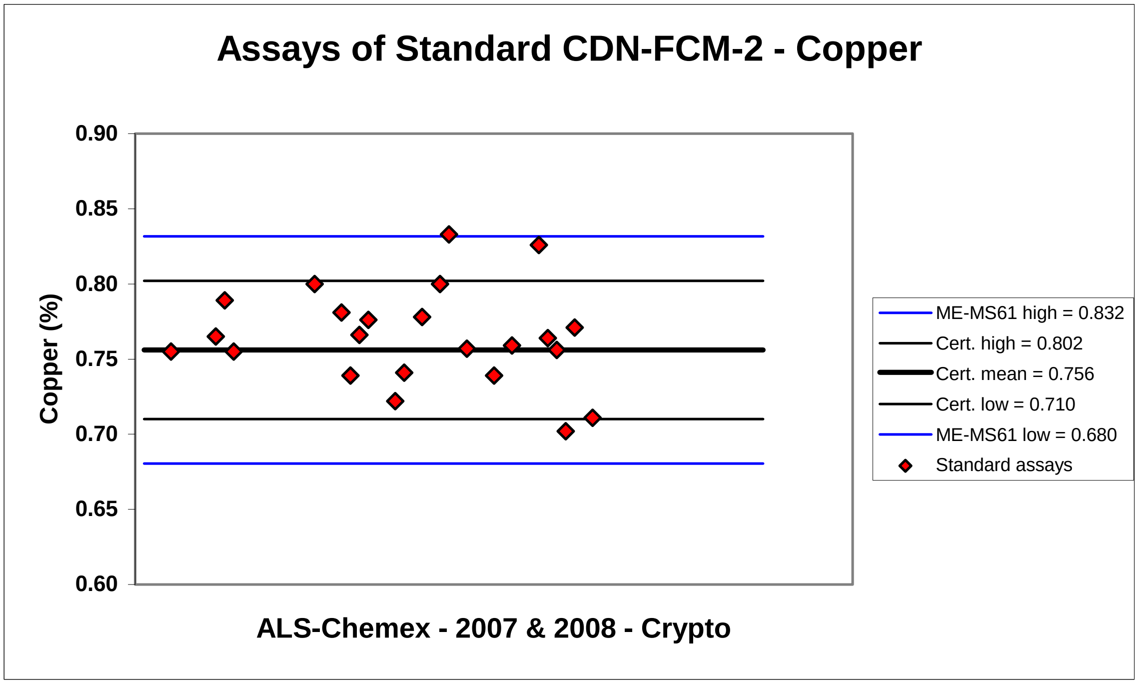


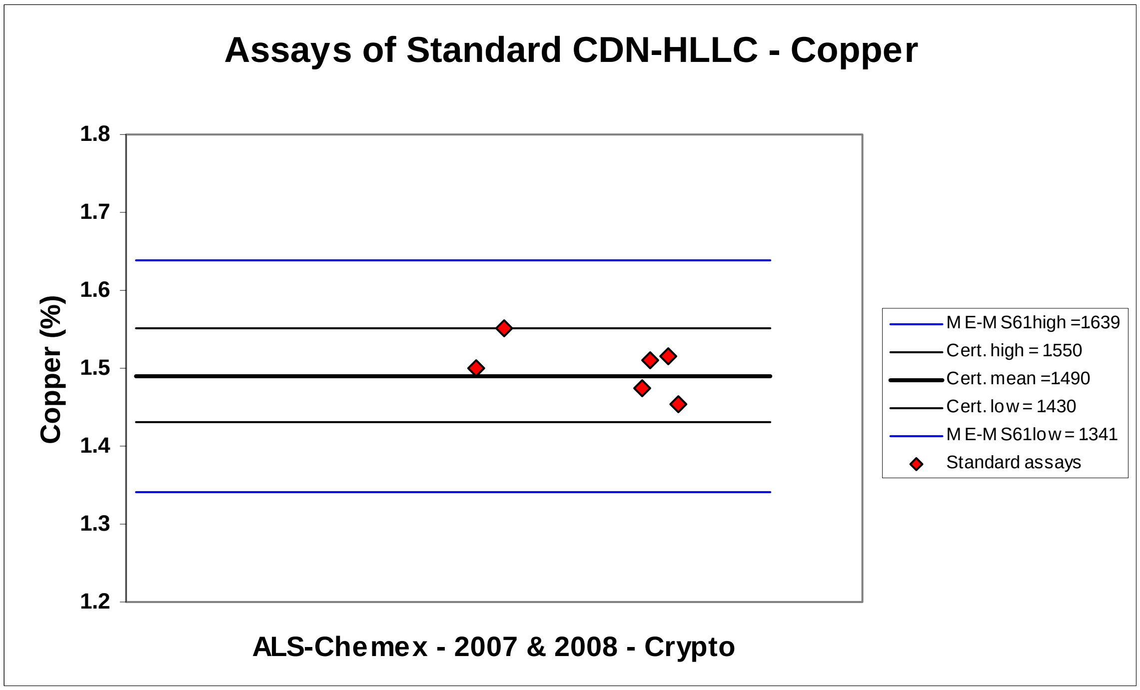

Assays of Standard CDN-HLLC - Silver
90
80
70
60
50
40
Silver (ppm)
Cert. high = 71.8
ME-MS61 high =71.6
Cert. mean =65.1
ME-MS61 low = 58.6
Cert. low = 58.4
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto

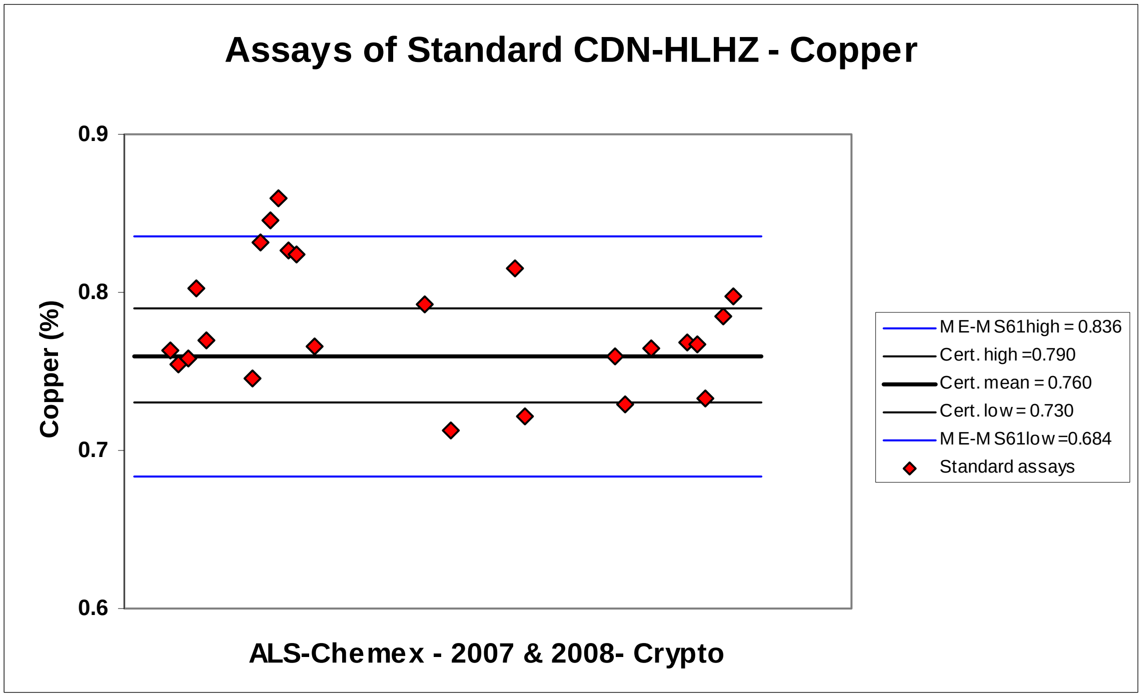

Assays of Standard CDN-HLHZ - Silver
120
110
100
90
80
Silver (g/t)
Cert. high = 112.0
ME-MS61 high = 111.3
Cert. mean = 101.2
ME-MS61 low = 91.1
Cert. low = 90.4
Standard assays
ALS-Chemex - 2007 & 2008- Crypto

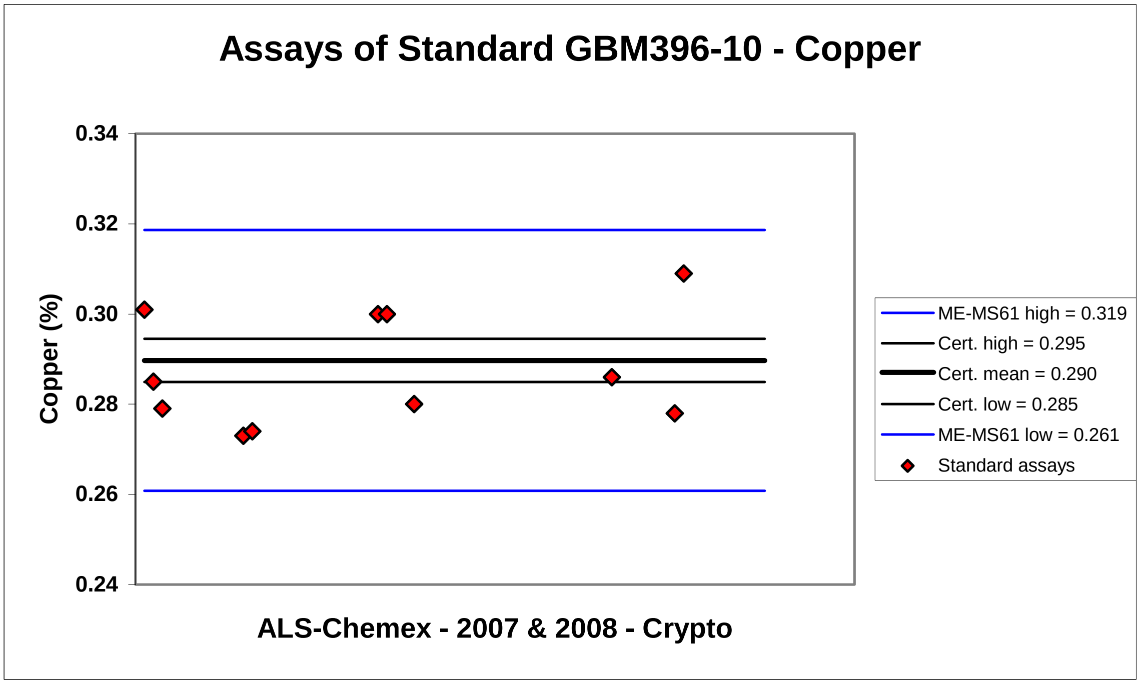

Assays of Standard GBM396-10 - Silver
14.00
12.00
10.00
8.00
Silver (ppm)
ALS-Chemex - 2007 & 2008 - Crypto
ME-MS61 high = 12.78
Cert. high = 11.82
Cert. mean = 11.60
Cert. low = 11.38
ME-MS61 low = 10.42
Standard assays

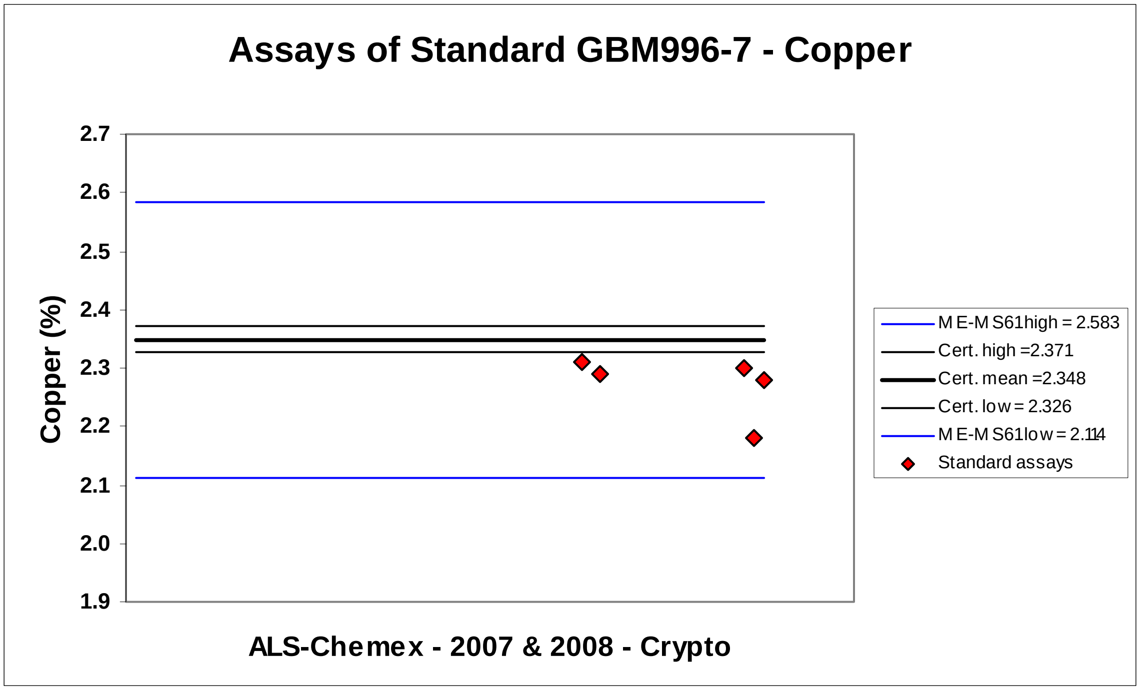

Assays of Standard GBM996-7 - Silver
150.0
140.0
130.0
120.0
110.0
100.0
Silver (ppm)
ALS-Chemex - 2007 & 2008 - Crypto
ME-MS61 high = 137.63
Cert. high =127.22
Cert. mean =125.10
Cert. low = 122.98
ME-MS61 low = 112.57
Standard assays

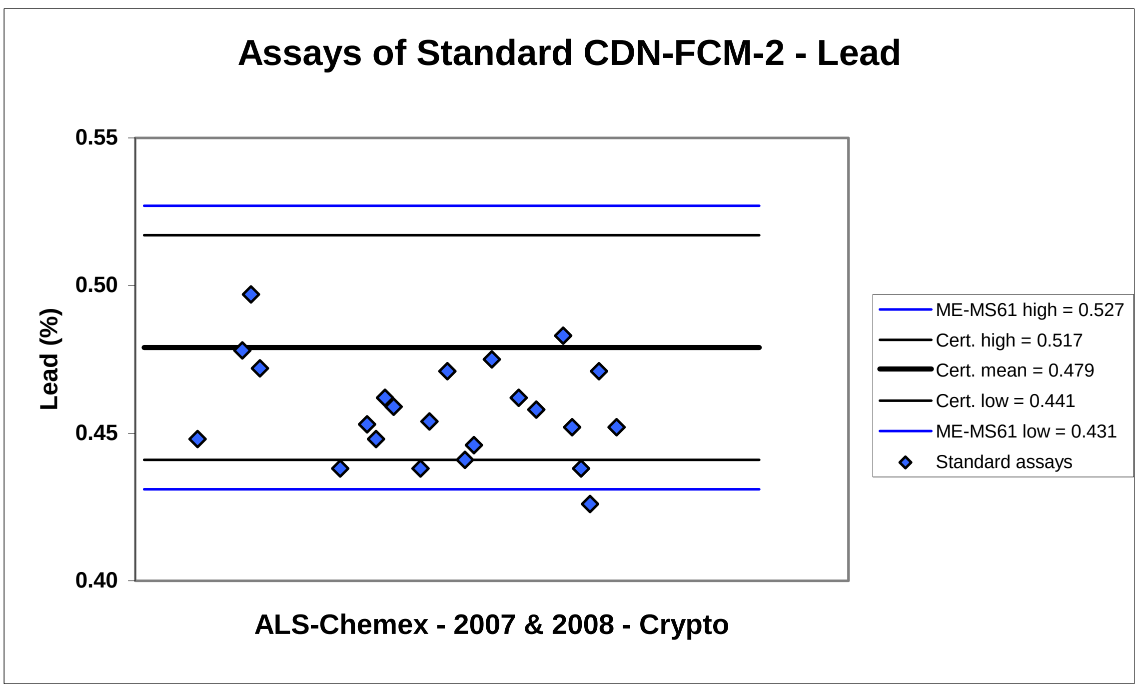

Assays of Standard CDN-FCM-2 - Copper
0.90
0.85
0.80
0.75
0.70
0.65
0.60
Copper (%)
ALS-Chemex - 2007 & 2008 - Crypto
ME-MS61 high = 0.832
Cert. high = 0.802
Cert. mean = 0.756
Cert. low = 0.710
ME-MS61 low = 0.680
Standard assays

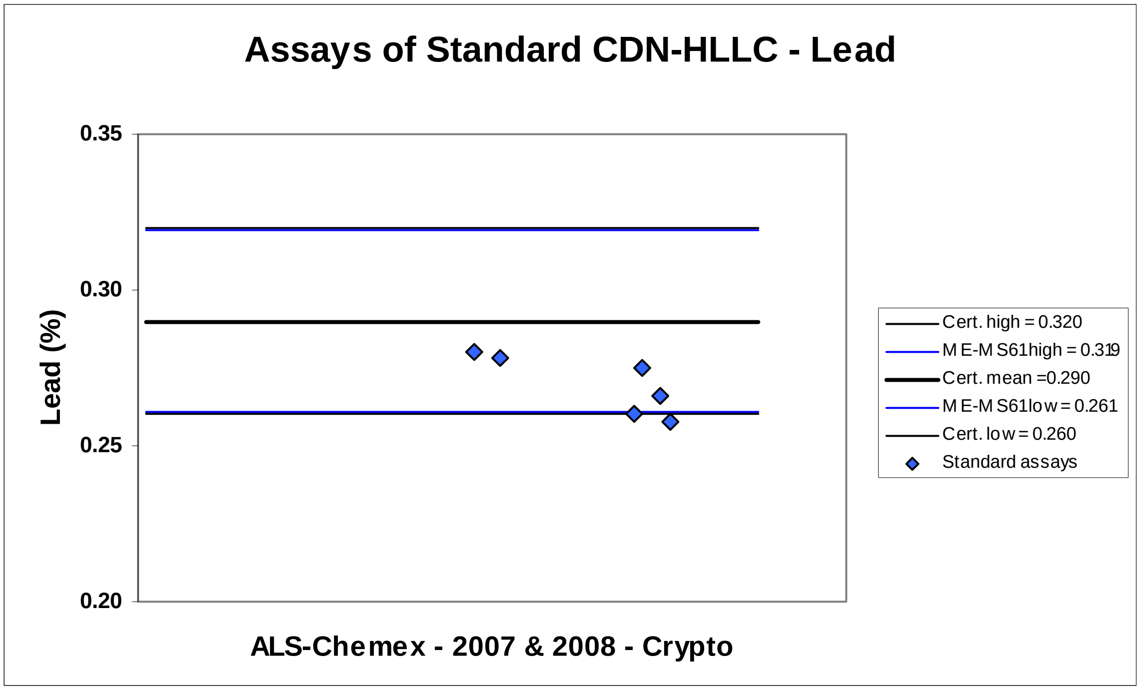

Assays of Standard CDN-HLLC - Copper
1.8
1.7
1.6
1.5
1.4
1.3
1.2
Copper (%)
ALS-Chemex - 2007 & 2008 - Crypto
ME-MS61high =1639
Cert. high = 1550
Cert. mean =1490
Cert. low = 1430
ME-MS61low = 1341
Standard assays

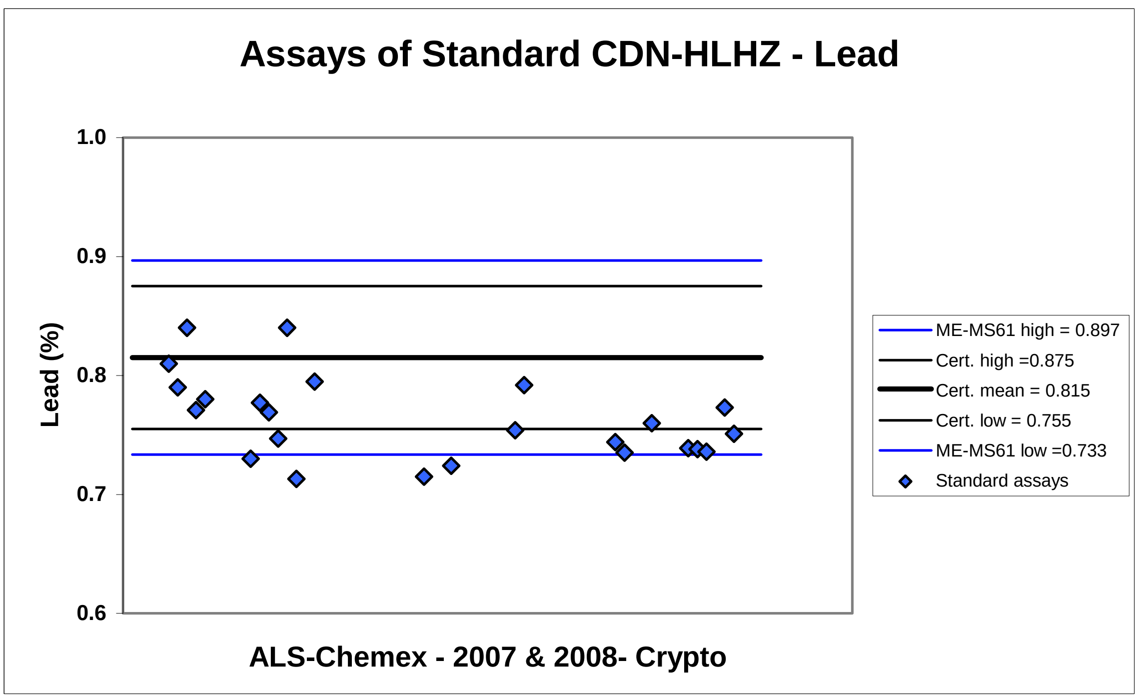

Assays of Standard CDN-HLHZ - Copper
0.9
0.8
0.7
0.6
Copper (%)
ME-MS61high = 0.836
Cert. high =0.790
Cert. mean = 0.760
Cert. low = 0.730
ME-MS61low =0.684
Standard assays
ALS-Chemex - 2007 & 2008- Crypto

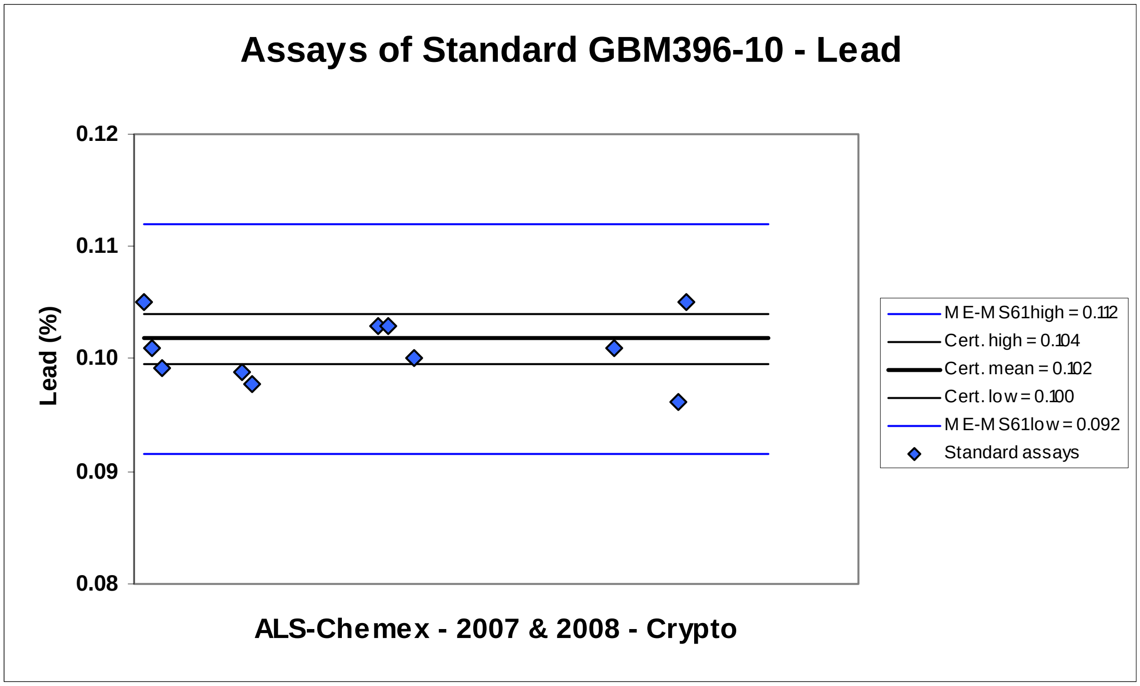

Assays of Standard GBM396-10 - Copper
0.34
0.32
0.30
0.28
0.26
0.24
Copper (%)
ME-MS61 high = 0.319
Cert. high = 0.295
Cert. mean = 0.290
Cert. low = 0.285
ME-MS61 low = 0.261
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto


Assays of Standard GBM996-7 - Copper
Copper (%)
2.7
2.6
2.5
2.4
2.3
2.2
2.1
2.0
1.9
ME-MS61high = 2.583
Cert. high =2.371
Cert. mean =2.348
Cert. low = 2.326
ME-MS61low = 2.114
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto

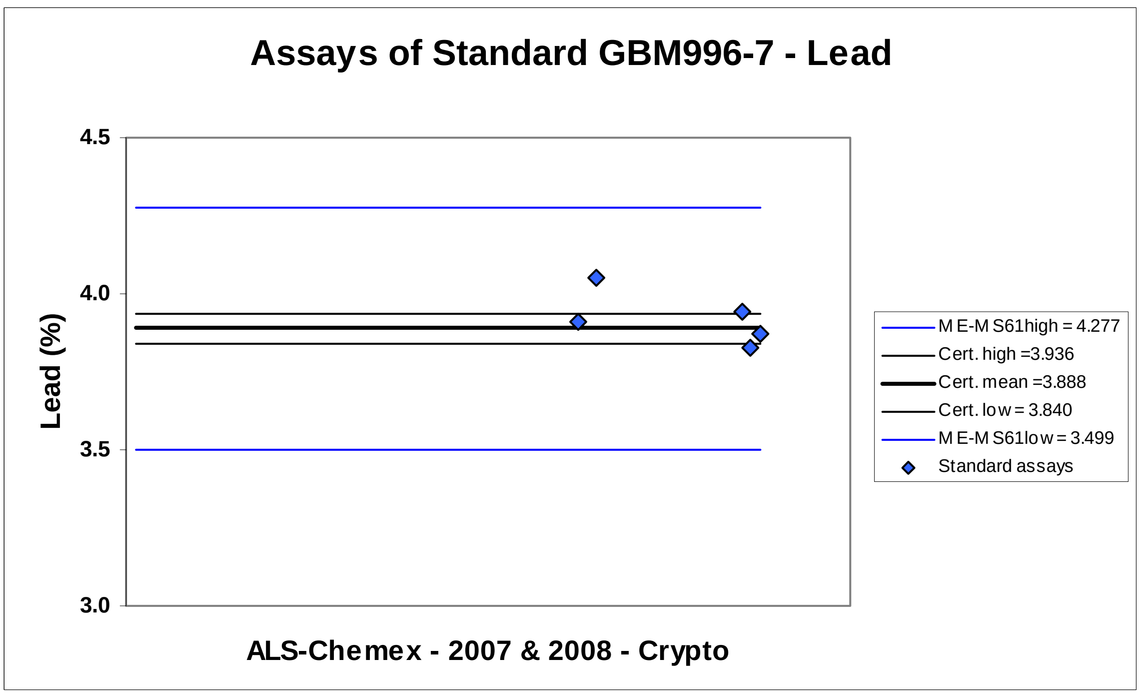

Assays of Standard CDN-FCM-2 - Lead
Lead (%)
0.55
0.50
0.45
0.40
ME-MS61 high = 0.527
Cert. high = 0.517
Cert. mean = 0.479
Cert. low = 0.441
ME-MS61 low = 0.431
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto


Assays of Standard CDN-HLLC - Lead
0.35
0.30
0.25
0.20
Lead (%)
Cert. high = 0.320
ME-MS61high = 0.319
Cert. mean =0.290
ME-MS61low = 0.261
Cert. low = 0.260
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto

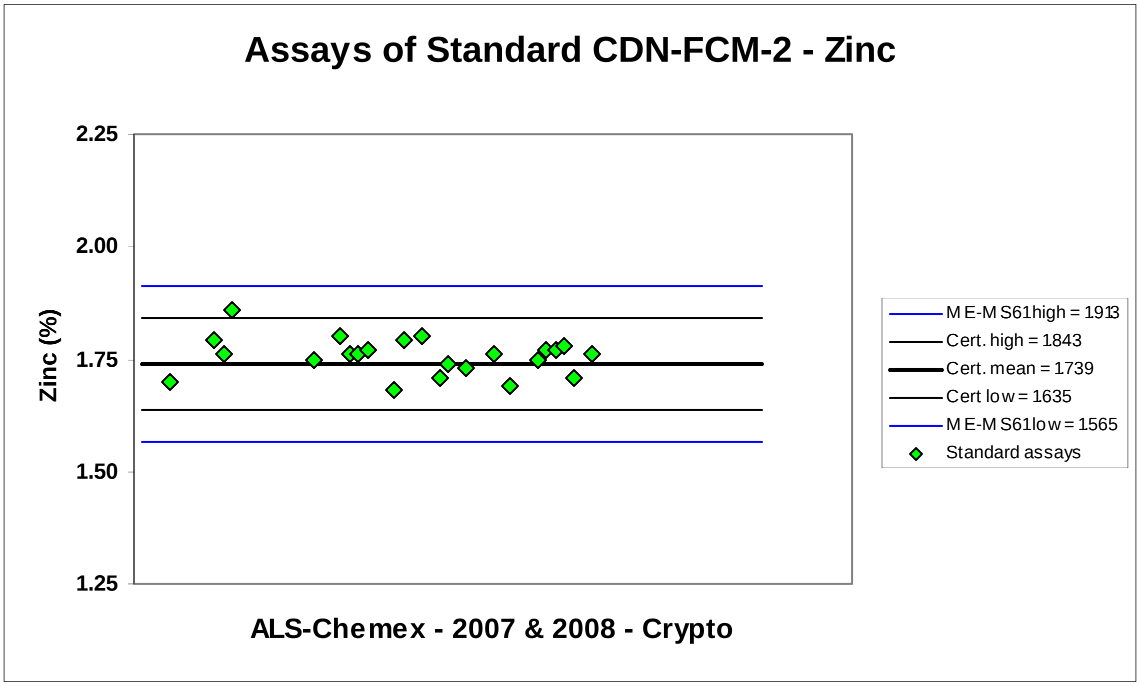

Assays of Standard CDN-HLHZ - Lead
1.0
0.9
0.8
0.7
0.6
Lead (%)
ME-MS61 high = 0.897
Cert. high =0.875
Cert. mean = 0.815
Cert. low = 0.755
ME-MS61 low =0.733
Standard assays
ALS-Chemex - 2007 & 2008- Crypto

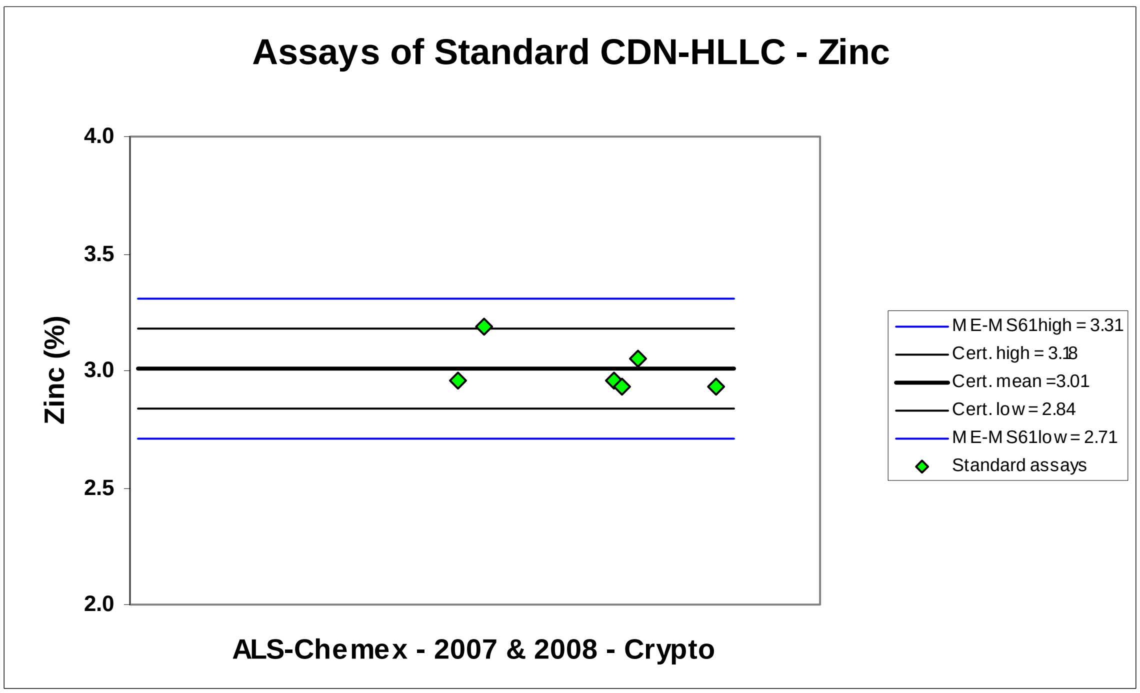


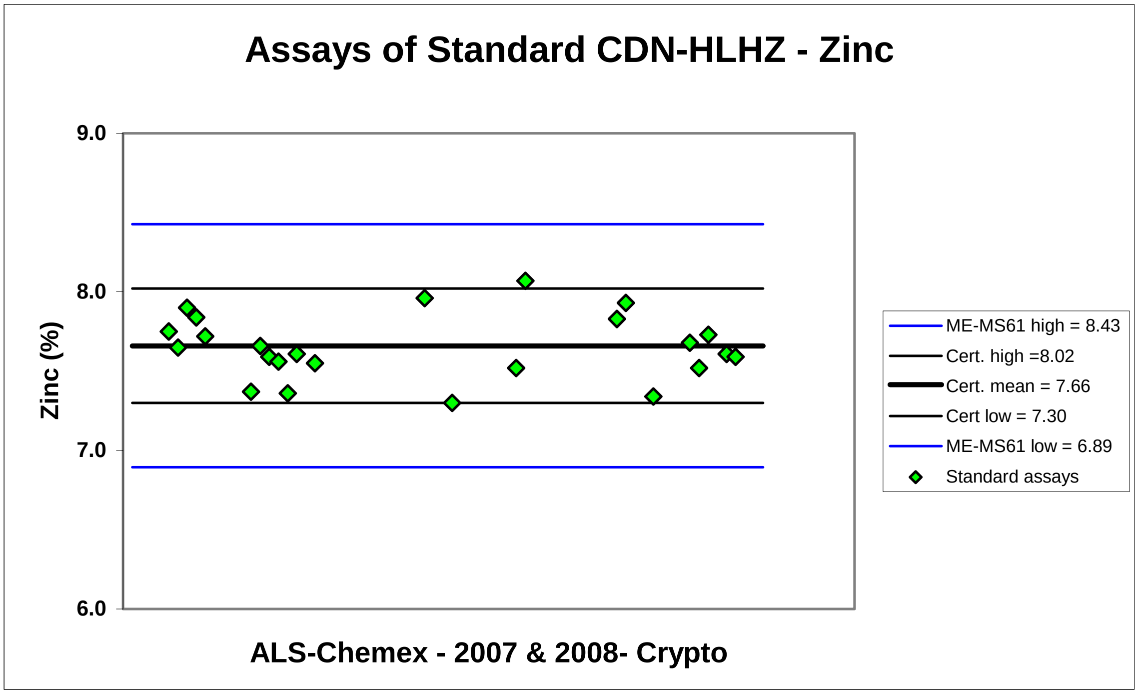
Assays of Standard GBM396-10 - Lead
Lead (%)
0.12
0.11
0.10
0.09
0.08
ALS-Chemex - 2007 & 2008 - Crypto
ME-MS61high = 0.112
Cert. high = 0.104
Cert. mean = 0.102
Cert. low = 0.100
ME-MS61low = 0.092
Standard assays



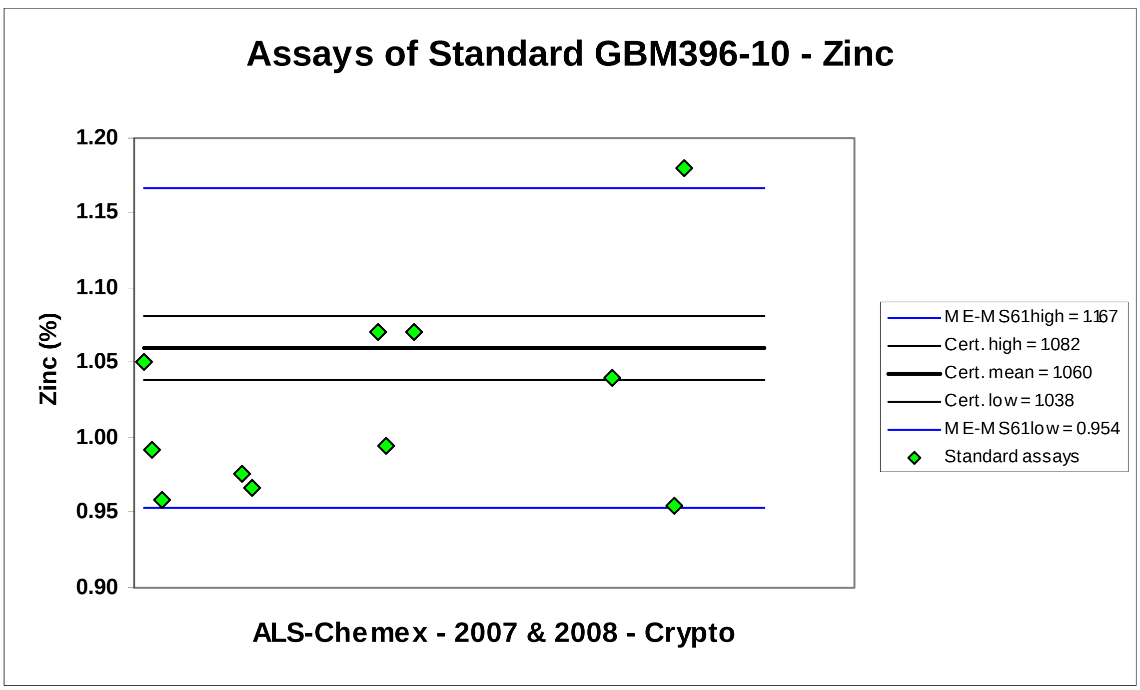
Assays of Standard GBM996-7 - Lead
Lead (%)
4.5
4.0
3.5
3.0
ALS-Chemex - 2007 & 2008 - Crypto
ME-MS61high = 4.277
Cert. high =3.936
Cert. mean =3.888
Cert. low = 3.840
ME-MS61low = 3.499
Standard assays


Assays of Standard CDN-FCM-2 - Zinc
2.25
2.00
1.75
1.50
1.25
Zinc (%)
ME-MS61high = 1913
Cert. high = 1843
Cert. mean = 1739
Cert low = 1635
ME-MS61low = 1565
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto

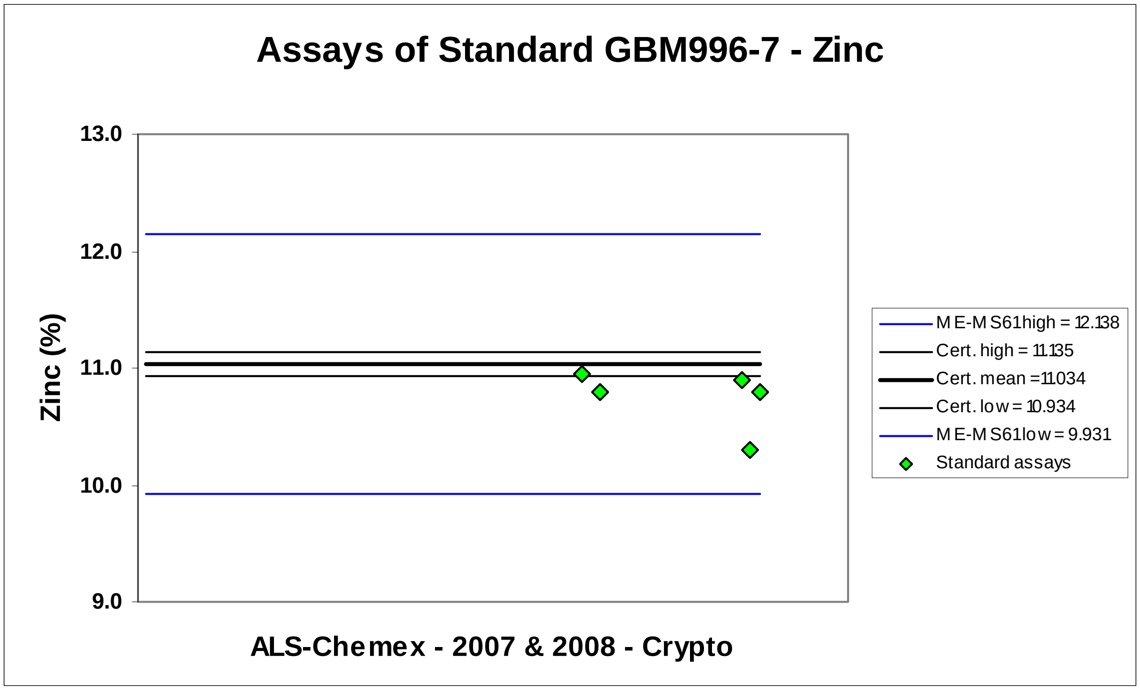

Assays of Standard CDN-HLLC - Zinc
4.0
3.5
3.0
2.5
2.0
Zinc (%)
ME-MS61high = 3.31
Cert. high = 3.18
Cert. mean =3.01
Cert. low = 2.84
ME-MS61low = 2.71
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto



Assays of Standard CDN-HLHZ - Zinc
9.0
8.0
7.0
6.0
Zinc (%)
ME-MS61 high = 8.43
Cert. high =8.02
Cert. mean = 7.66
Cert low = 7.30
ME-MS61 low = 6.89
Standard assays
ALS-Chemex - 2007 & 2008- Crypto



Assays of Standard GBM396-10 - Zinc
1.20
1.15
1.10
1.05
1.00
0.95
0.90
Zinc (%)
ME-MS61high = 1167
Cert. high = 1082
Cert. mean = 1060
Cert. low = 1038
ME-MS61low = 0.954
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto



These plots show that the performance of assays for Lithic inserted standards for the 2007 and 2008 core drilling at Crypto was in general acceptable. A very small number of results lie outside acceptable limits, but not far enough outside to be of serious concern.

Monitoring plots for un-certified elements – Lithic inserted standards, 2007 & 2008:

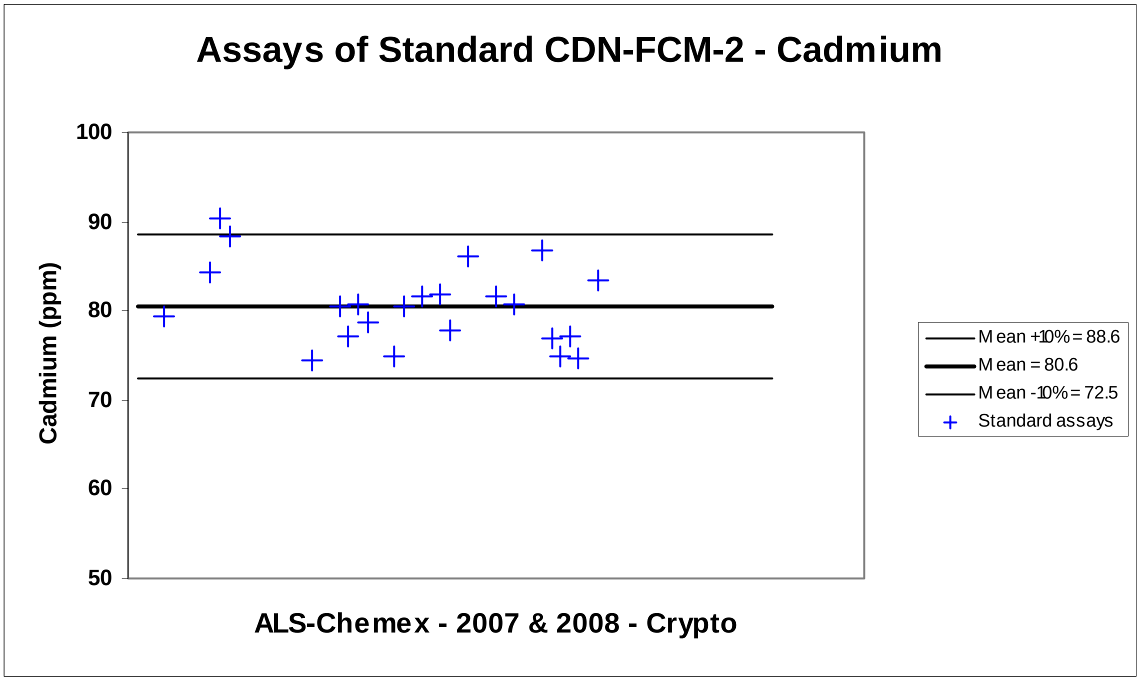
For the 2007 and 2008 drill programs, there are also analyses of the same five standards for four trace elements of interest: cadmium, gallium, germanium and indium.

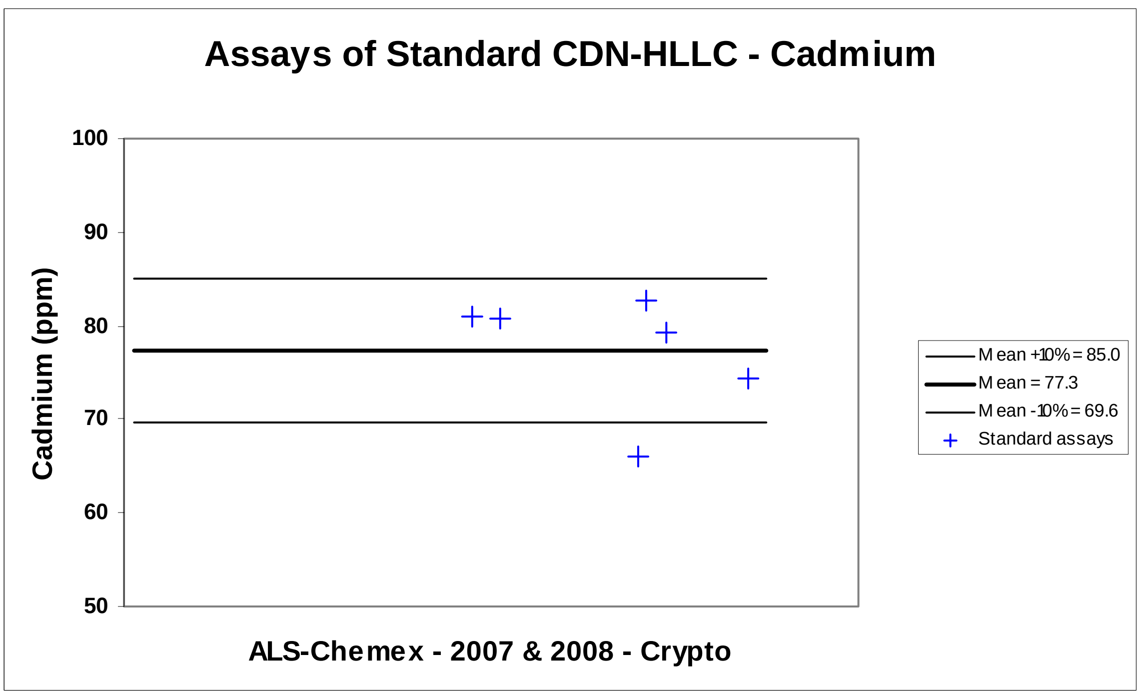
An attempt has been made to show the assay performance for these elements, for the standards in question. The plots on the following pages show the mean ± 10% values for the total number of analyses for that particular element for each standard.

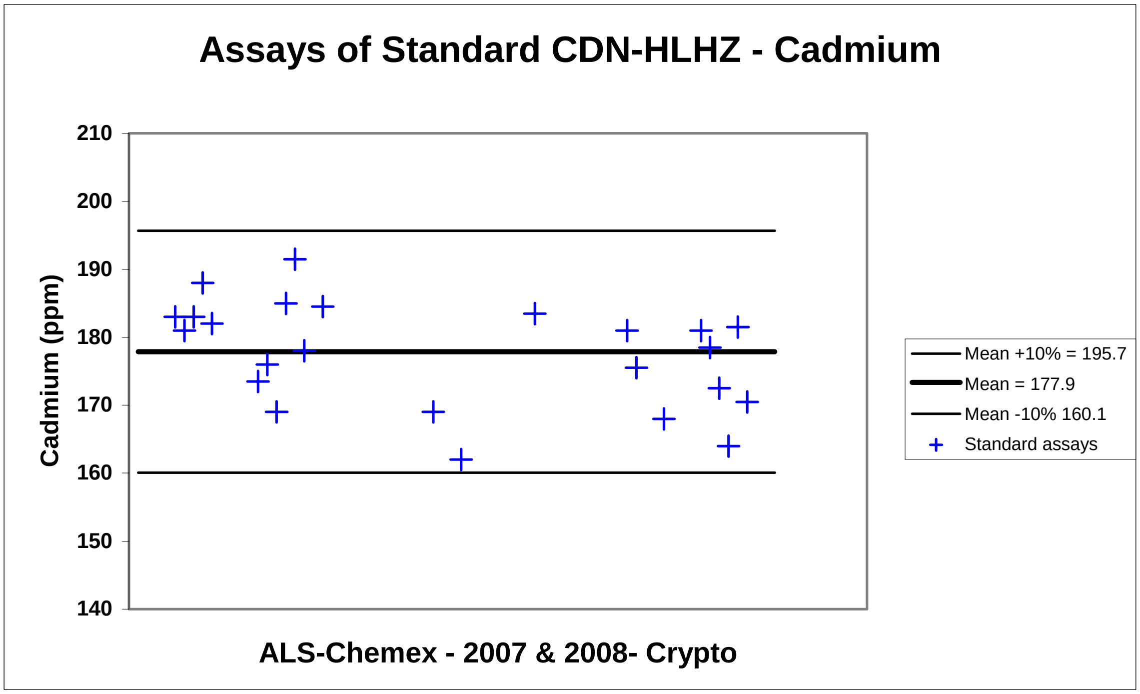
Clearly, these plots do not speak to accuracy, because there are no certified values to set the accepted levels. Rather, they are a very rough measure of the precision of assays for these elements. The problem is compounded by the fact that there are only a limited number of assays available to calculate the respective means.

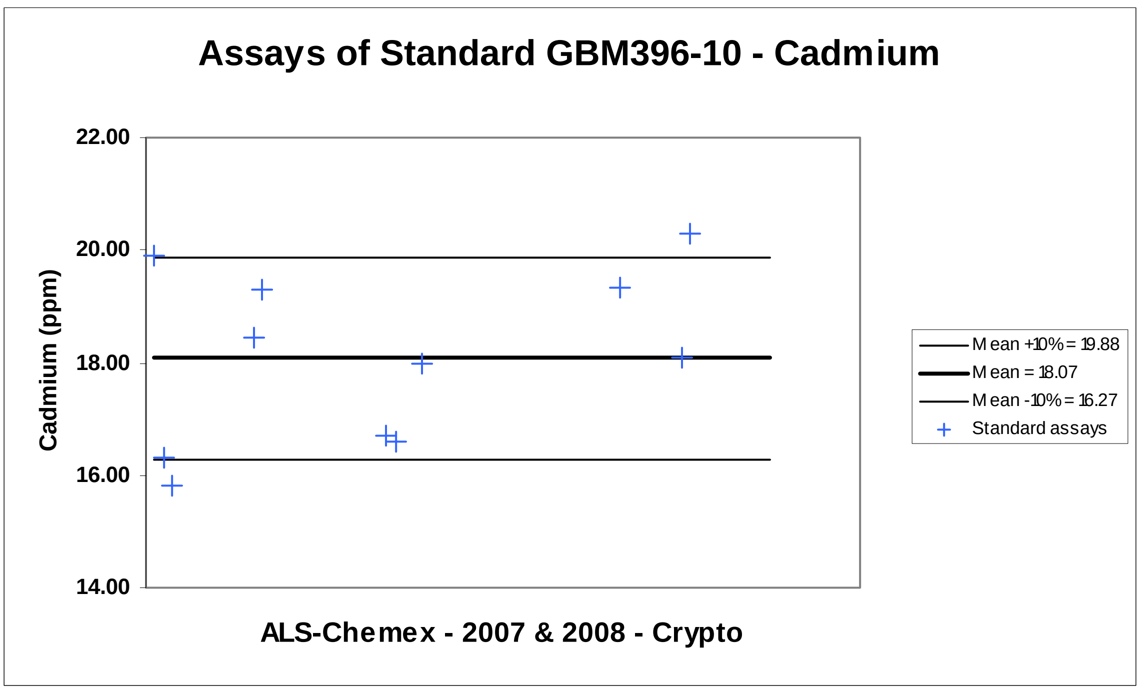
Charts for molybdenum have not been included, as the values in the standards are very low. All five standards have mean values for molybdenum, admittedly in some cases for very few analyses, of less than 100 parts per million. Almost all analyses lie within the ± 10 percent limits.

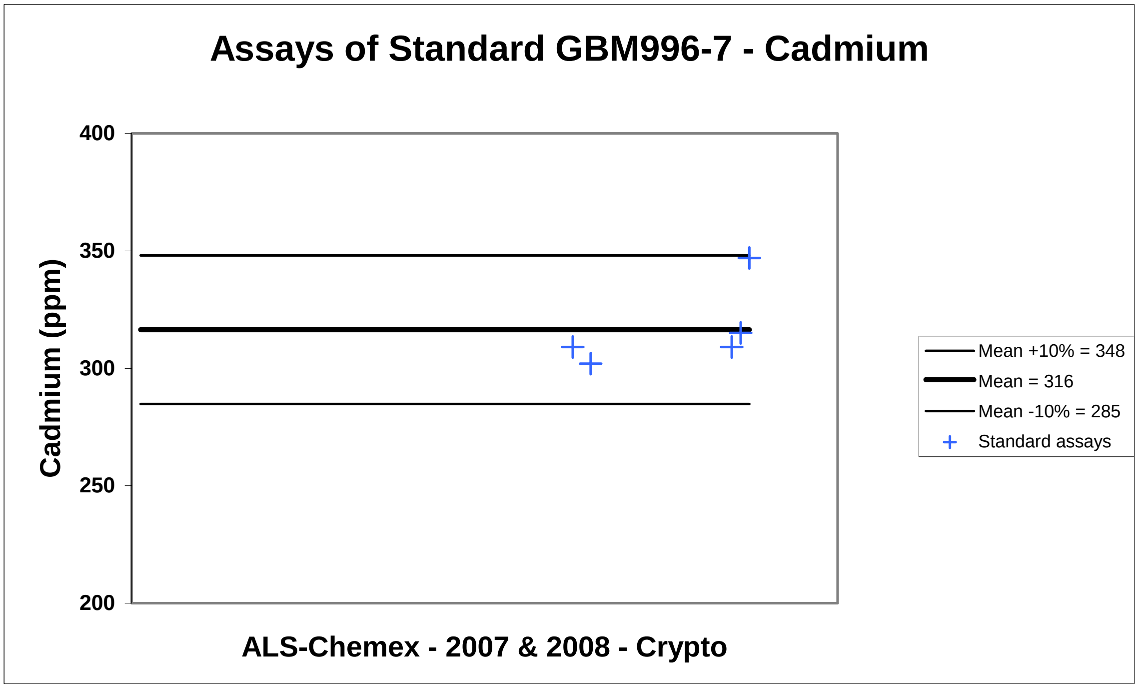
There were insufficient data for gold to warrant preparing charts for this metal.

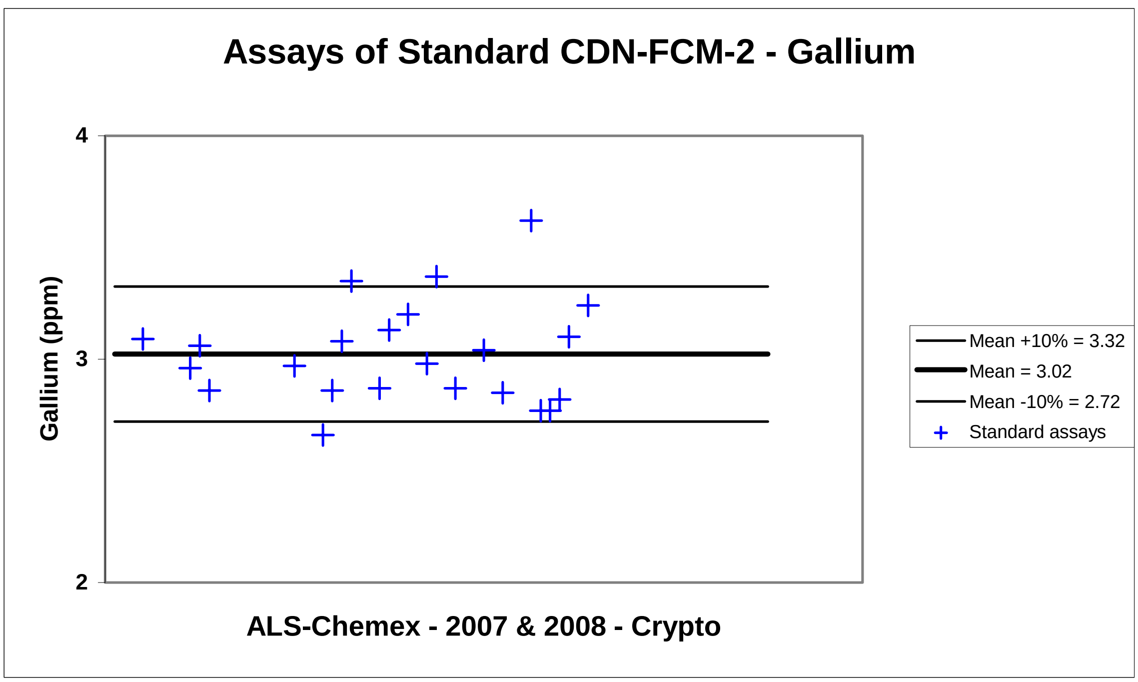

Assays of Standard CDN-FCM-2 - Cadmium
Cadmium (ppm)
100
90
80
70
60
50
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 88.6
Mean = 80.6
Mean -10% = 72.5
Standard assays

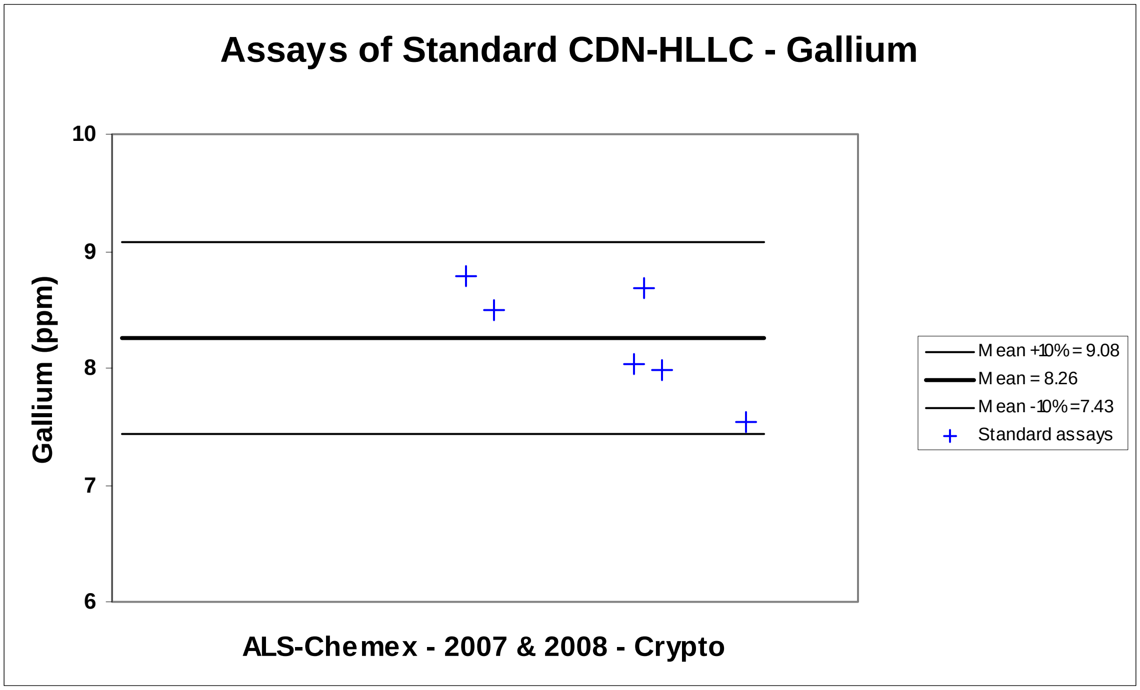

Assays of Standard CDN-HLLC - Cadmium
Cadmium (ppm)
100
90
80
70
60
50
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 85.0
Mean = 77.3
Mean -10% = 69.6
Standard assays

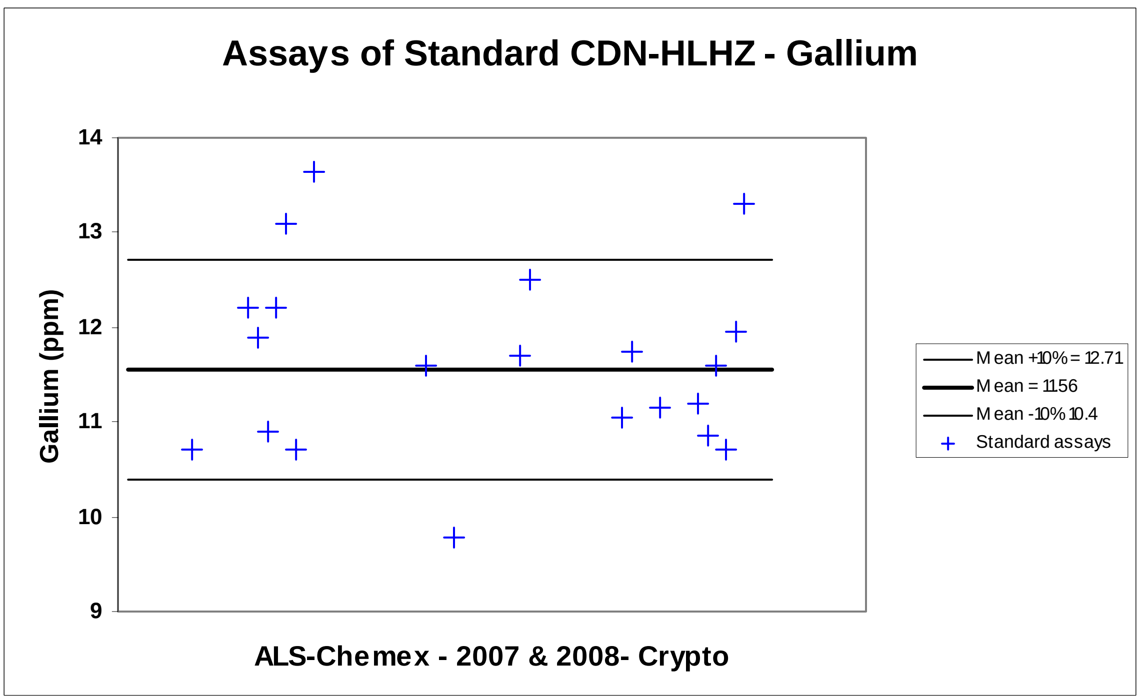

Assays of Standard CDN-HLHZ - Cadmium
210
200
190
180
170
160
150
140
Cadmium (ppm)
Mean +10% = 195.7
Mean = 177.9
Mean -10% 160.1
Standard assays
ALS-Chemex - 2007 & 2008- Crypto

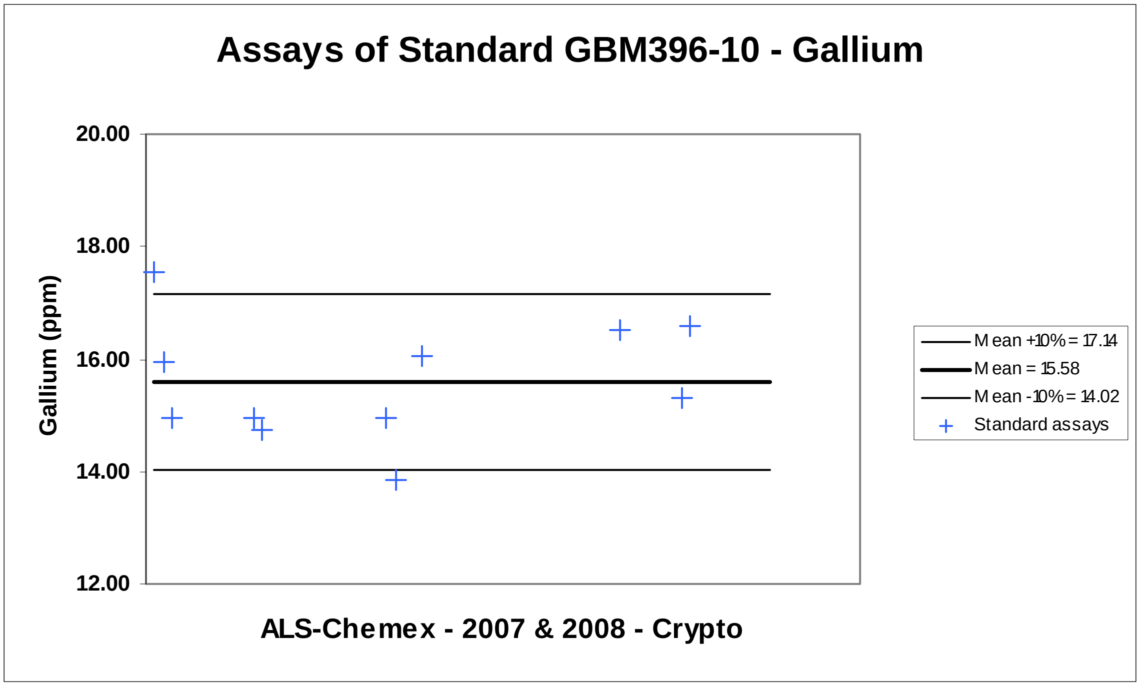

Assays of Standard GBM396-10 - Cadmium
22.00
20.00
18.00
16.00
14.00
Cadmium (ppm)
Mean +10% = 19.88
Mean = 18.07
Mean -10% = 16.27
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto

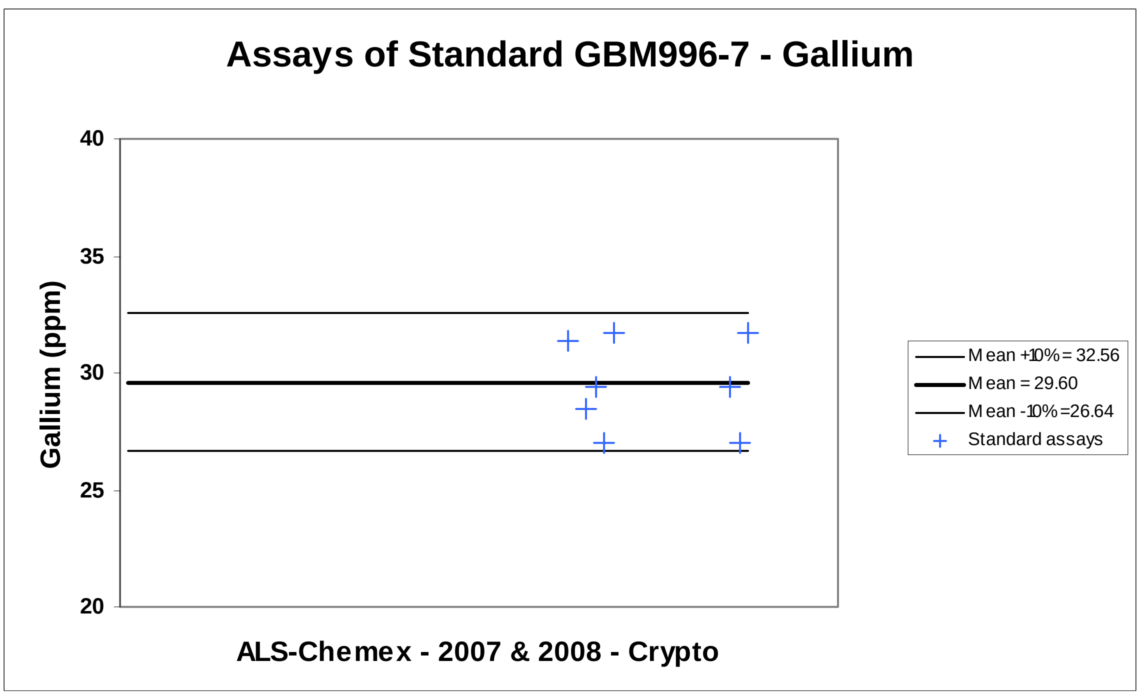

Assays of Standard GBM996-7 - Cadmium
Cadmium (ppm)
400
350
300
250
200
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 348
Mean = 316
Mean -10% = 285
Standard assays

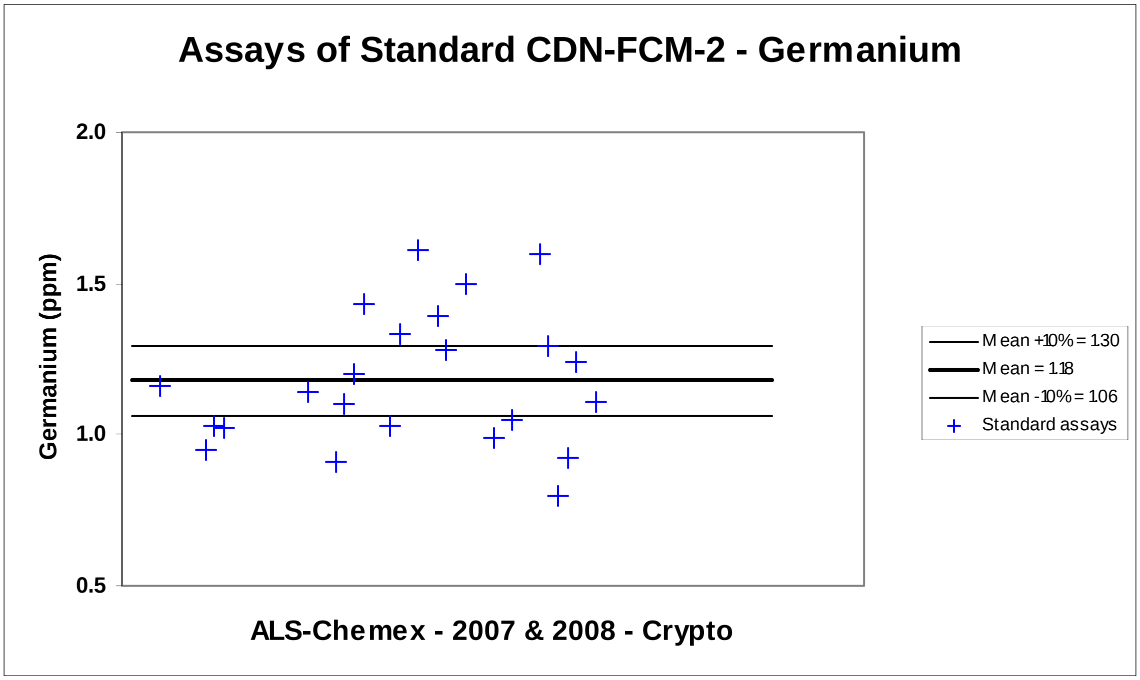

Assays of Standard CDN-FCM-2 - Gallium
Gallium (ppm)
4
3
2
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 3.32
Mean = 3.02
Mean -10% = 2.72
Standard assays

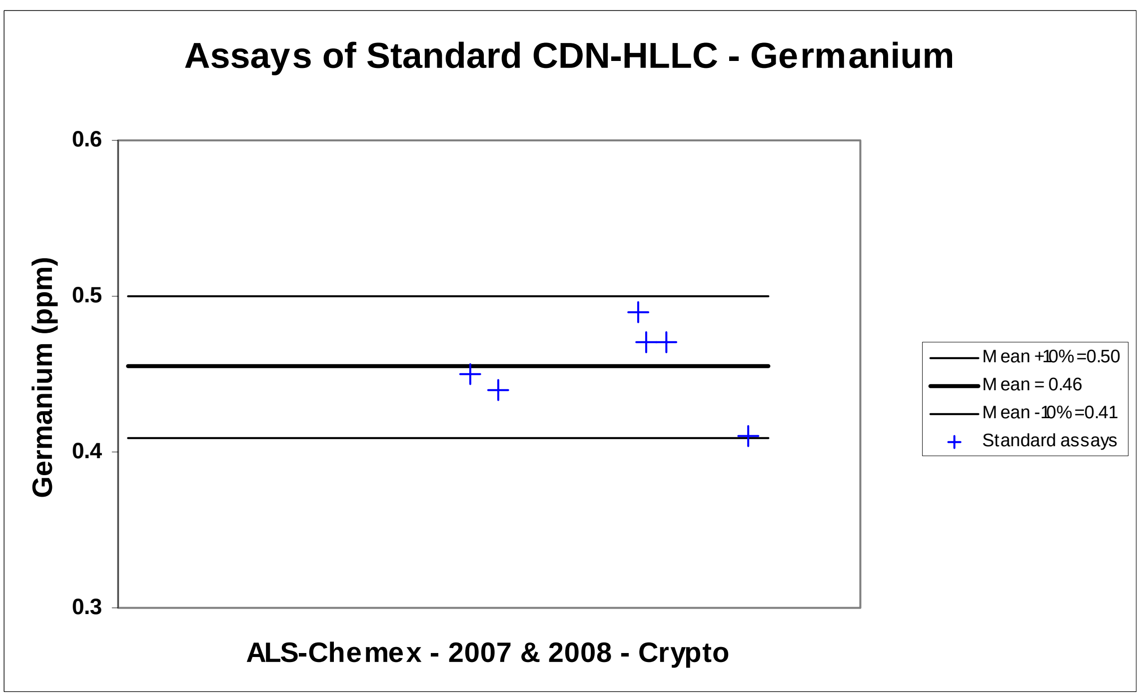

Assays of Standard CDN-HLLC - Gallium
Gallium (ppm)
10
9
8
7
6
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 9.08
Mean = 8.26
Mean -10% =7.43
Standard assays

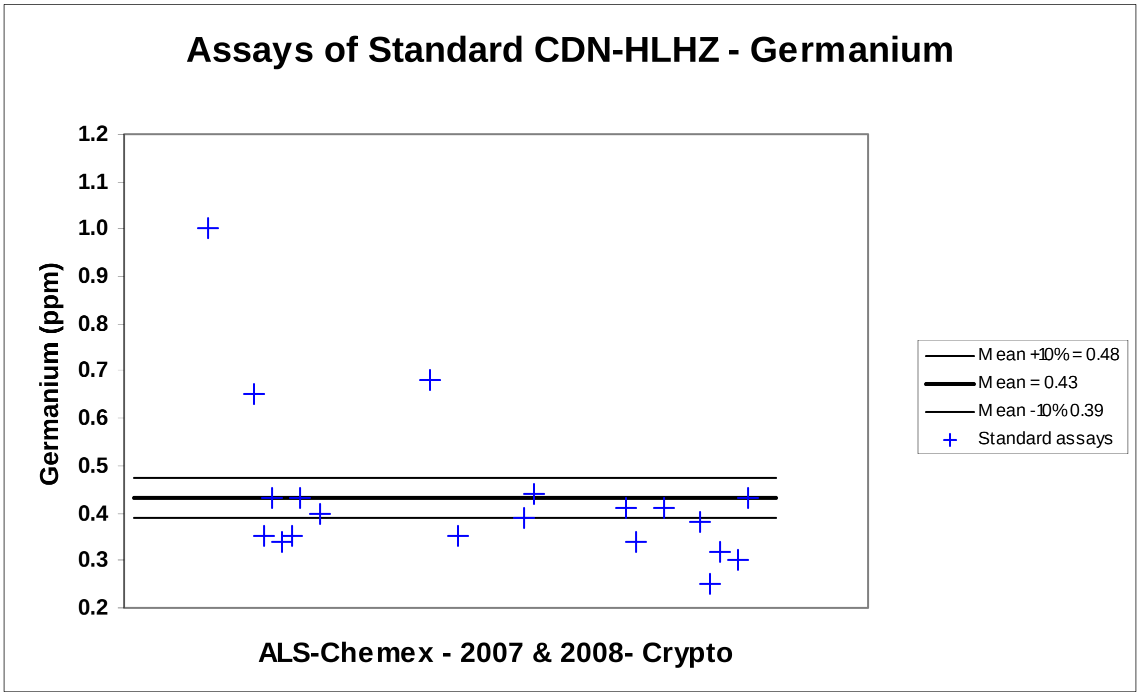

Assays of Standard CDN-HLHZ - Gallium
Gallium (ppm)
14
13
12
11
10
9
ALS-Chemex - 2007 & 2008- Crypto
Mean +10% = 12.71
Mean = 11.56
Mean -10% 10.4
Standard assays

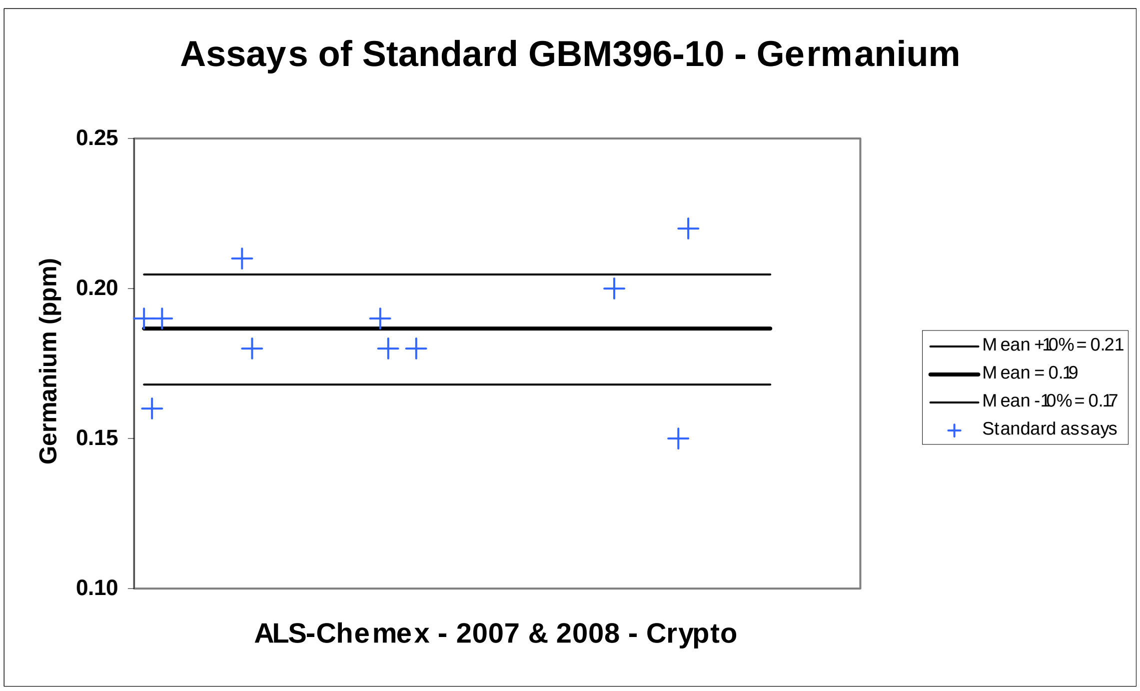

Assays of Standard GBM396-10 - Gallium
20.00
18.00
16.00
14.00
12.00
Gallium (ppm)
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 17.14
Mean = 15.58
Mean -10% = 14.02
Standard assays

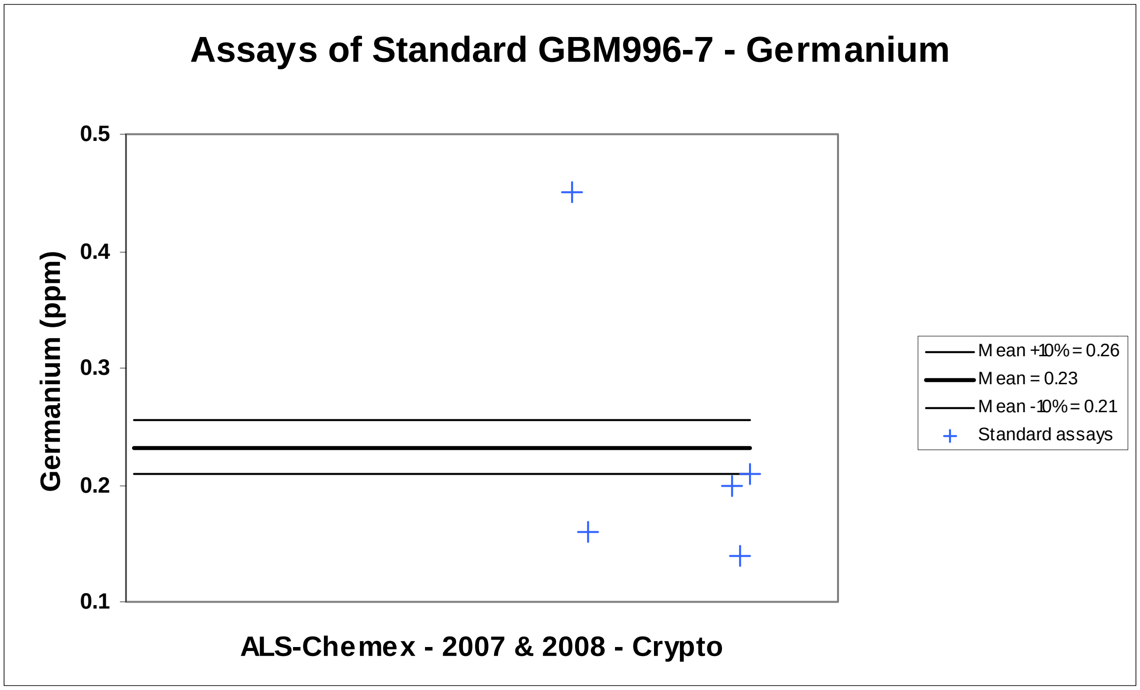

Assays of Standard GBM996-7 - Gallium
40
35
30
25
20
Gallium (ppm)
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 32.56
Mean = 29.60
Mean -10% =26.64
Standard assays

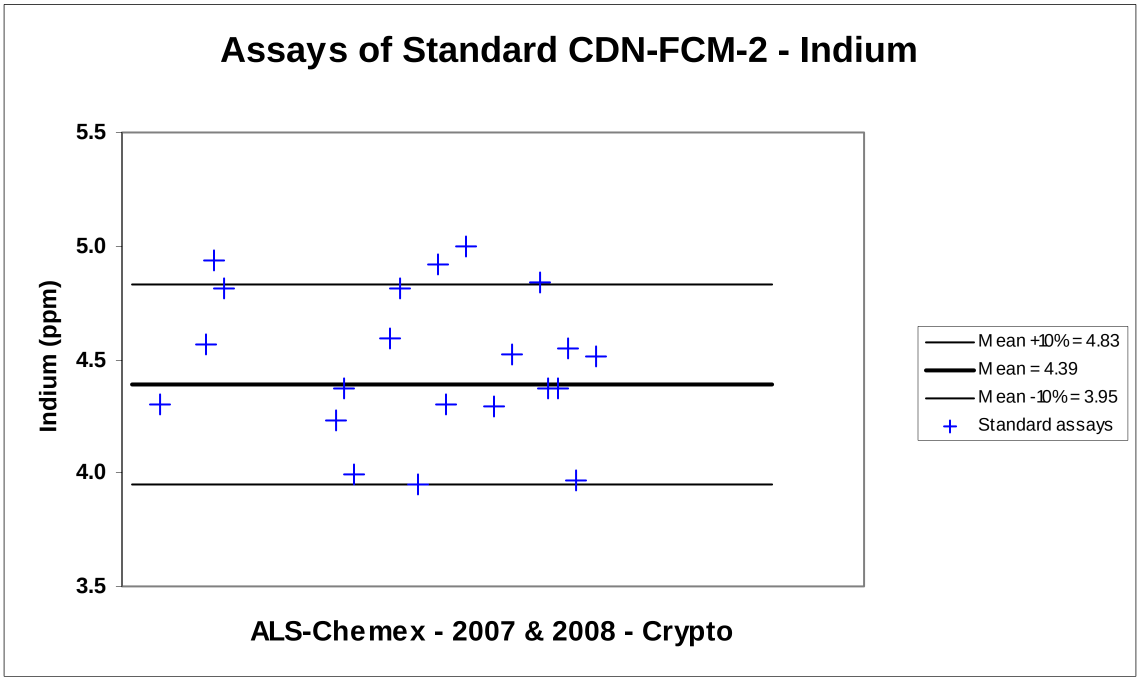

Assays of Standard CDN-FCM-2 - Germanium
2.0
1.5
1.0
0.5
Germanium (ppm)
Mean +10% = 1.30
Mean = 1.18
Mean -10% = 1.06
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto

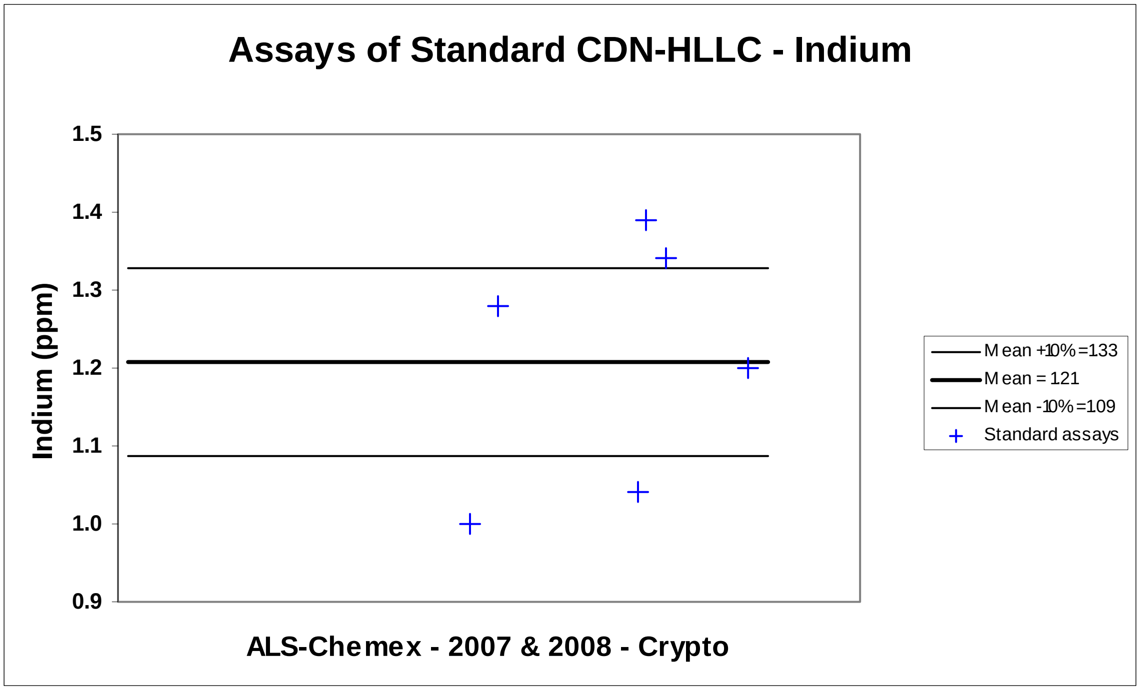

Assays of Standard CDN-HLLC - Germanium
0.6
0.5
0.4
0.3
Germanium (ppm)
Mean +10% =0.50
Mean = 0.46
Mean -10% =0.41
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto


Assays of Standard CDN-HLHZ - Germanium
Germanium (ppm)
1.2
1.1
1.0
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
ALS-Chemex - 2007 & 2008- Crypto
Mean +10% = 0.48
Mean = 0.43
Mean -10% 0.39
Standard assays

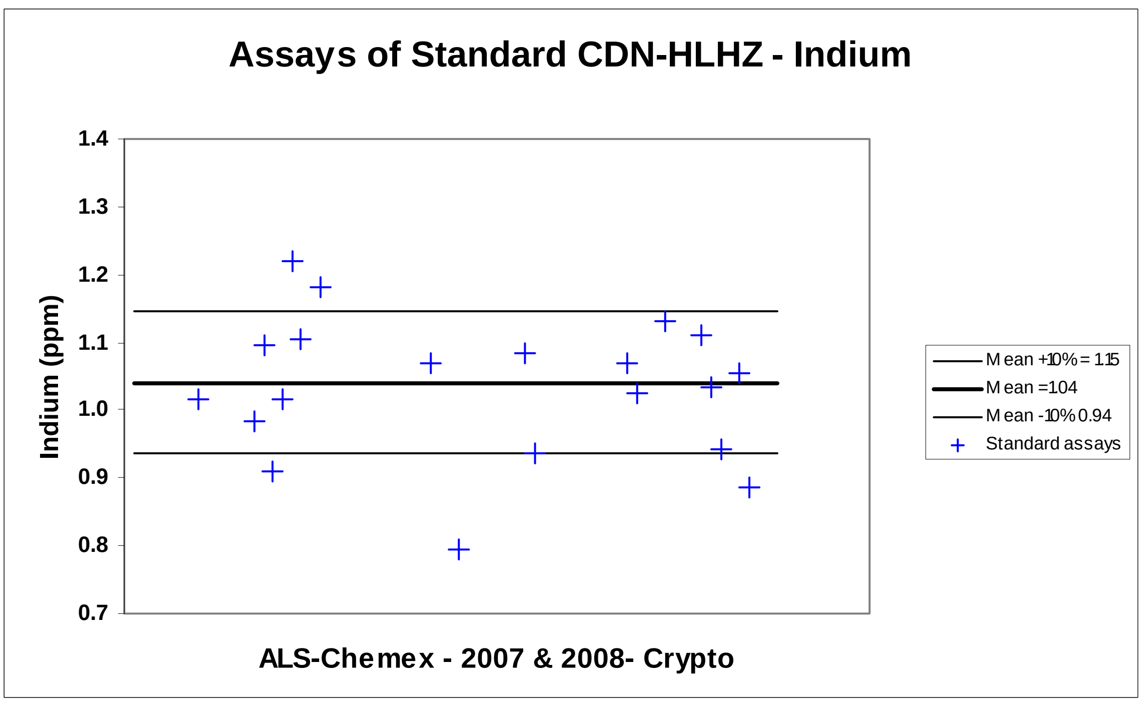

Assays of Standard GBM396-10 - Germanium
Germanium (ppm)
0.25
0.20
0.15
0.10
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 0.21
Mean = 0.19
Mean -10% = 0.17
Standard assays


Assays of Standard GBM996-7 - Germanium
Germanium (ppm)
0.5
0.4
0.3
0.2
0.1
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 0.26
Mean = 0.23
Mean -10% = 0.21
Standard assays

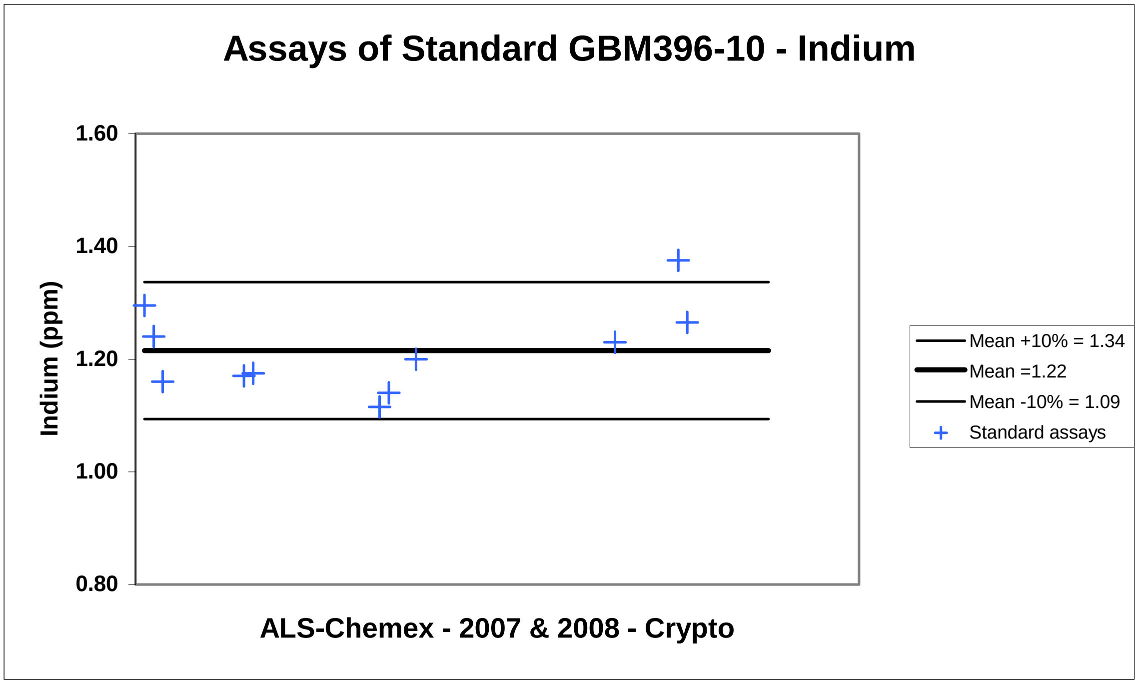

Assays of Standard CDN-FCM-2 - Indium
Indium (ppm)
5.5
5.0
4.5
4.0
3.5
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 4.83
Mean = 4.39
Mean -10% = 3.95
Standard assays

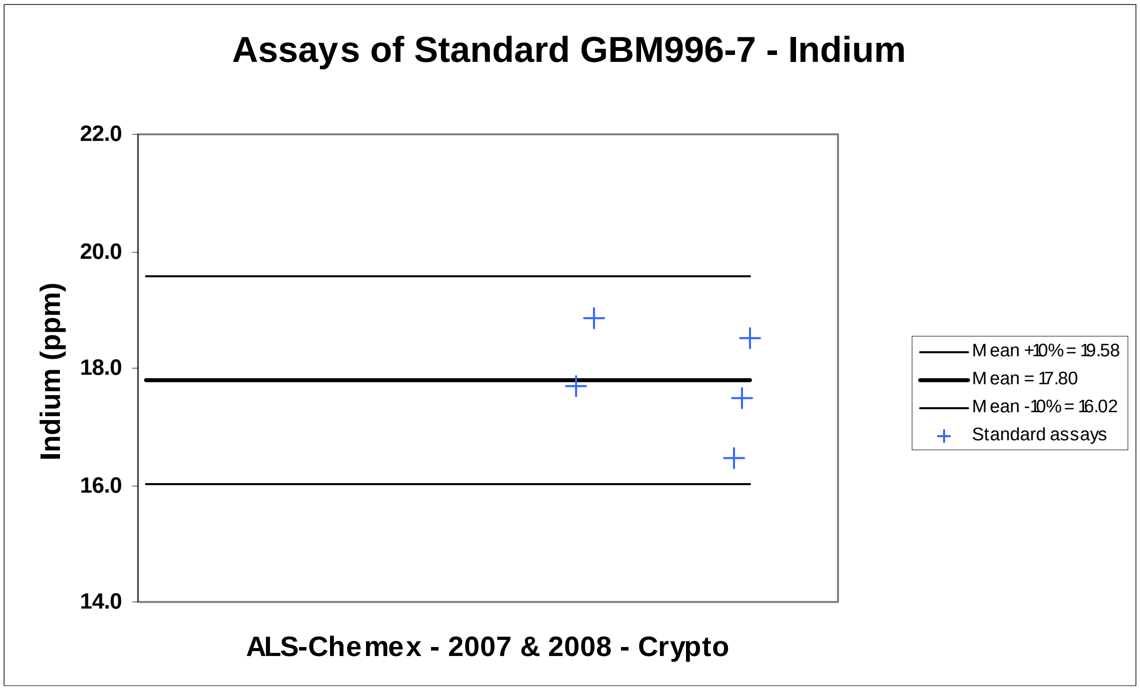


Assays of Standard CDN-HLLC - Indium
Indium (ppm)
1.5
1.4
1.3
1.2
1.1
1.0
0.9
Mean +10% = 1.33
Mean = 1.21
Mean -10% = 1.09
Standard assays
ALS-Chemex - 2007 & 2008 - Crypto



Assays of Standard CDN-HLHZ - Indium
Indium (ppm)
1.4
1.3
1.2
1.1
1.0
0.9
0.8
0.7
Mean +10% = 1.15
Mean = 1.04
Mean -10% 0.94
Standard assays
ALS-Chemex - 2007 & 2008- Crypto


Assays of Standard GBM396-10 - Indium
Indium (ppm)
1.60
1.40
1.20
1.00
0.80
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 1.34
Mean =1.22
Mean -10% = 1.09
Standard assays


Assays of Standard GBM996-7 - Indium
Indium (ppm)
22.0
20.0
18.0
16.0
14.0
ALS-Chemex - 2007 & 2008 - Crypto
Mean +10% = 19.58
Mean = 17.80
Mean -10% = 16.02
Standard assays

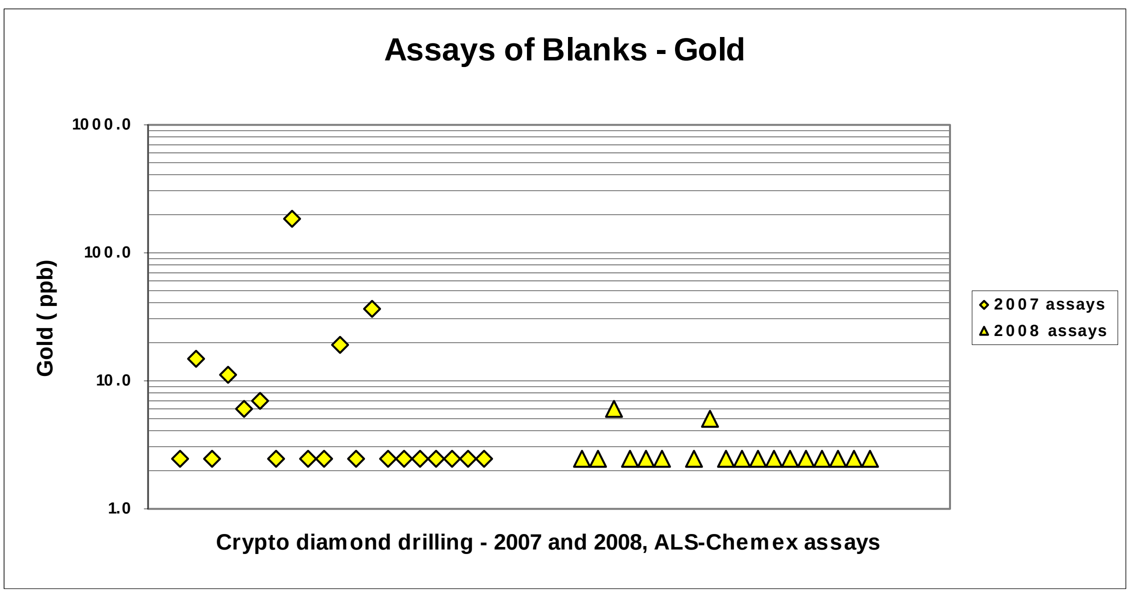
The results shown on these plots suggest that analyses for these four elements are less than optimally precise. Fortunately, those for indium are perhaps the best, which is encouraging as this is the trace element of most interest. Unfortunately, none of the standards has a mean in range of interest for indium; the highest value being 17.8 parts per million for standard GBM996-7.

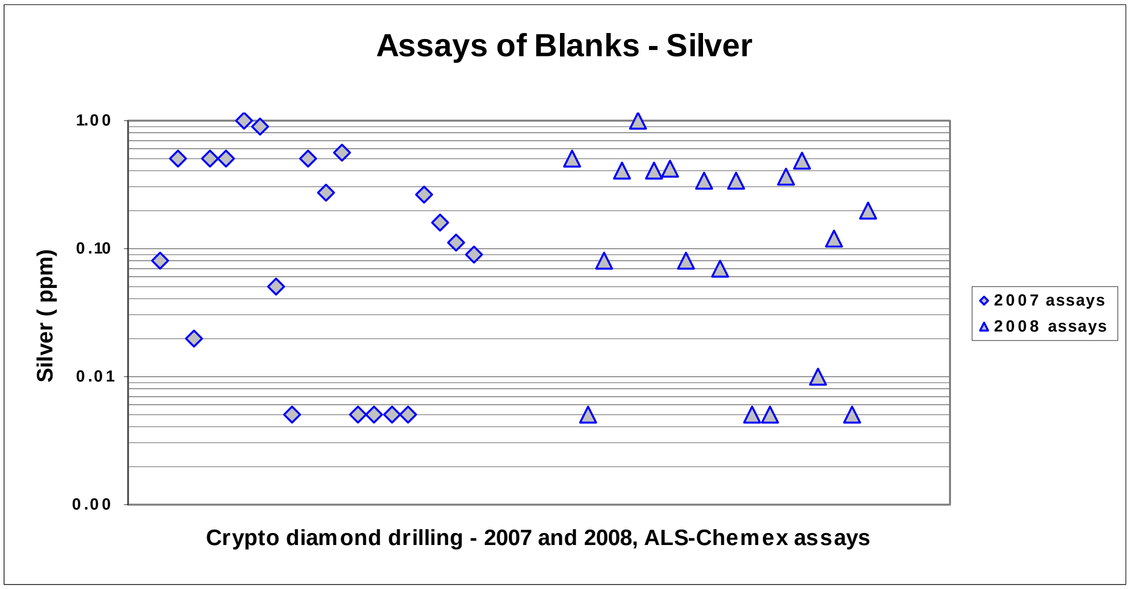
The overall conclusion must be that levels of these trace elements should be determined in concentrates during the metallurgical testing process, and that specific standards must be in place at that time.

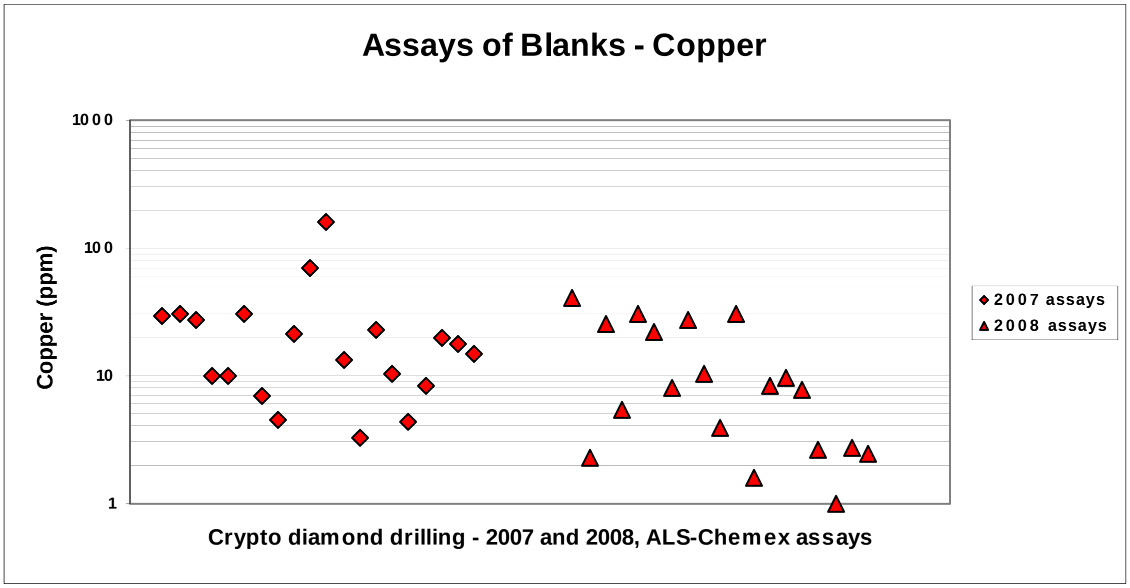
Monitoring plots – Lithic inserted "blanks", 2007 & 2008 drilling:

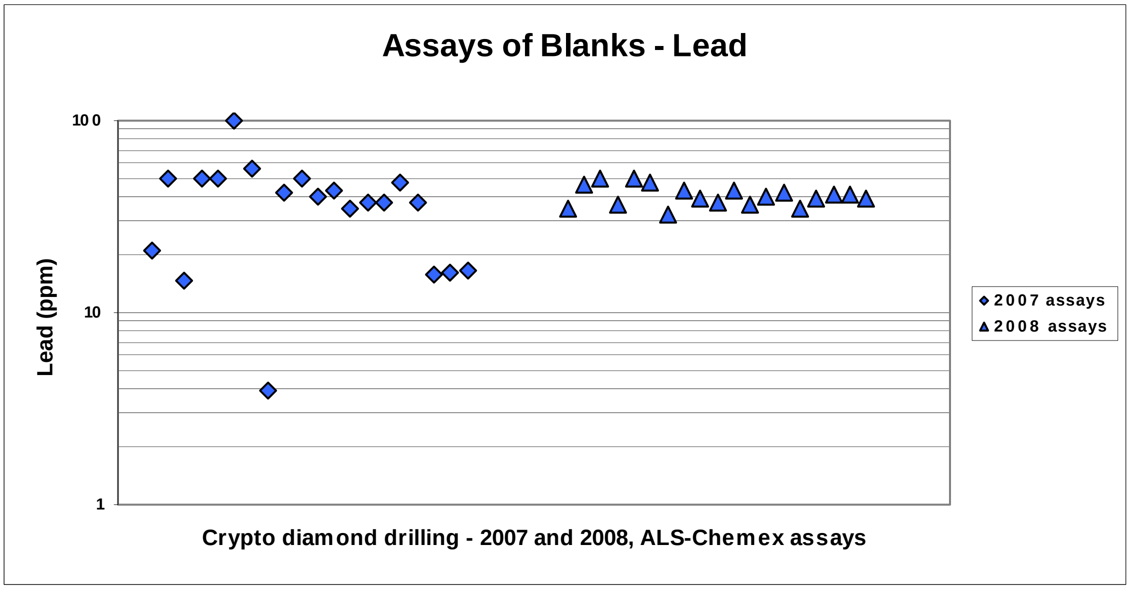
For the Crypto core drilling in 2008, coarse "blank" samples were included. Blanks are inserted to try to evaluate the possibility of contamination from previous samples in the preparation process. The material used was collected from a surface exposure of Tertiary rhyolite well removed from the property, assumed to have a negligible metal content. Assays to date lead one to question if this assumption is strictly speaking true, but on balance the results are satisfactory, with a few caveats.

Plots of the performance of blank assays for the two drilling phases are presented on the following pages. Note that the vertical axis scales are logarithmic.

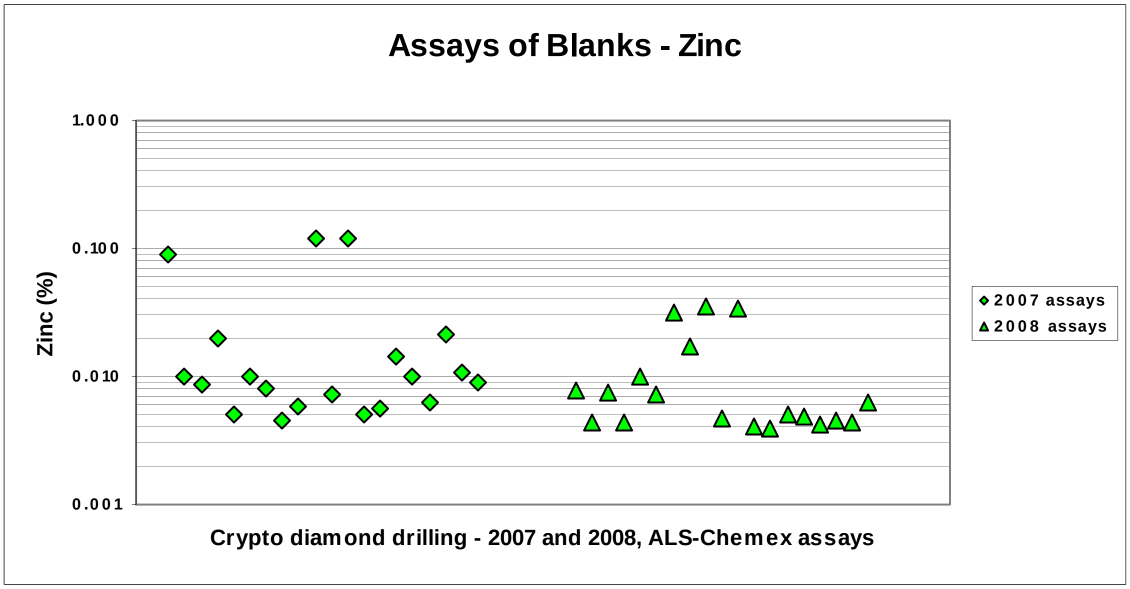



Assays of Blanks - Silver
Silver (ppm)
1.00
0.10
0.01
0.00
Crypto diamond drilling - 2007 and 2008, ALS-Chemex assays
2007 assays
2008 assays

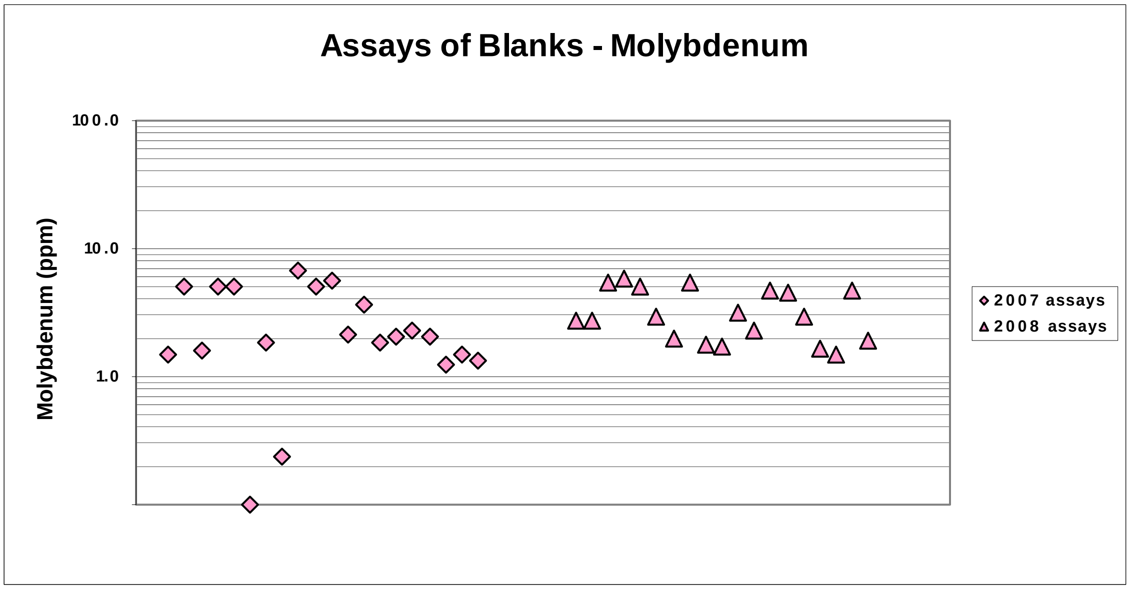

Assays of Blanks - Copper
Copper (ppm)
1000
100
10
1
Crypto diamond drilling - 2007 and 2008, ALS-Chemex assays
2007 assays
2008 assays

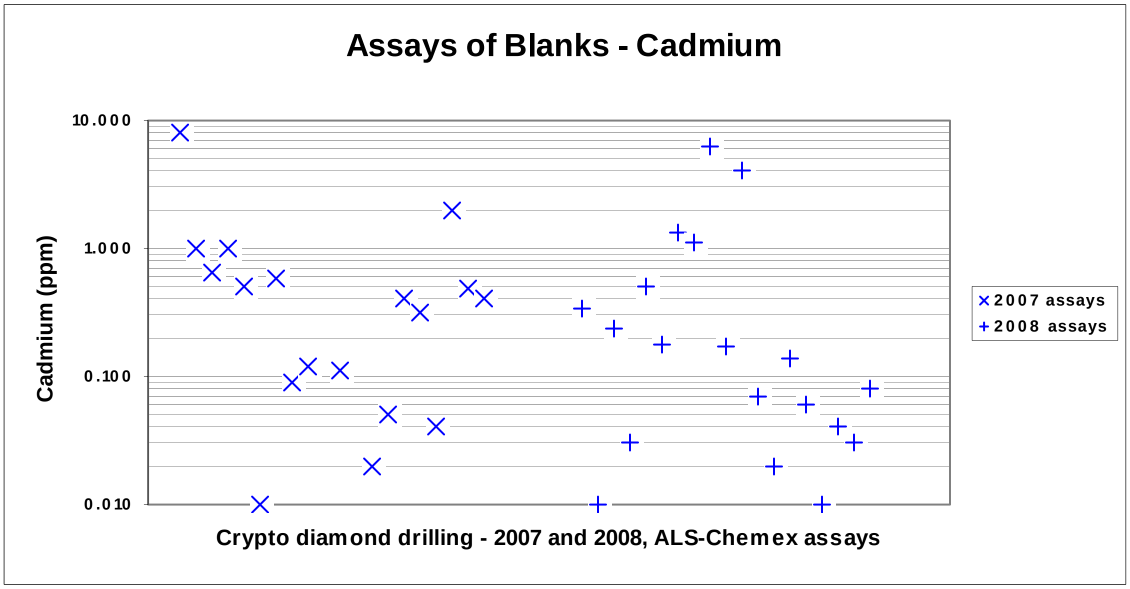


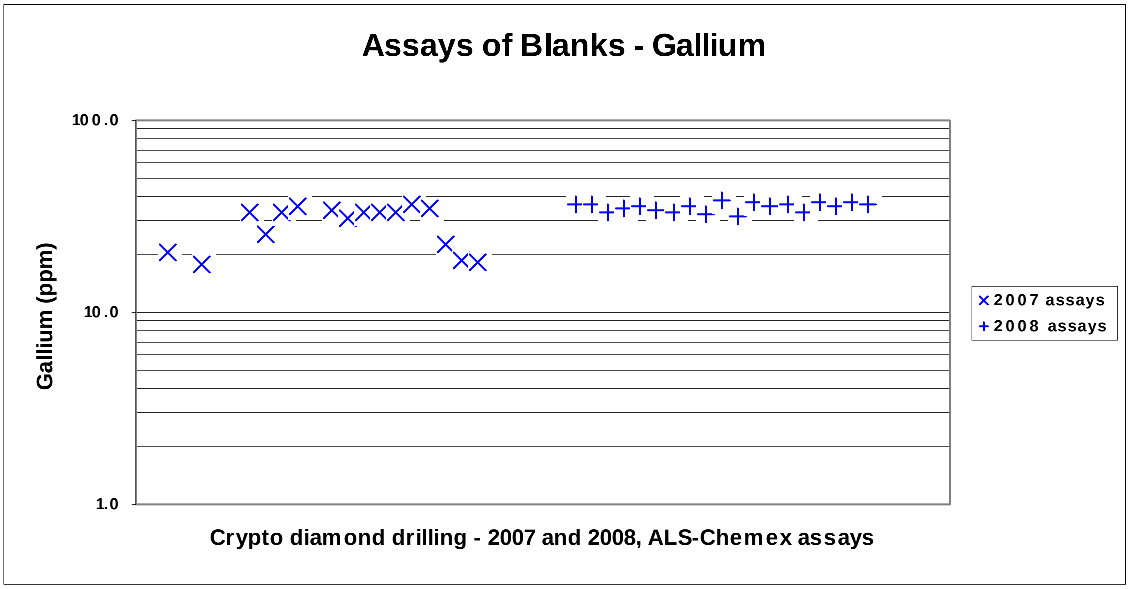
Assays of Blanks - Lead
100
10
1
Lead (ppm)
Crypto diamond drilling - 2007 and 2008, ALS-Chemex assays
2007 assays
2008 assays

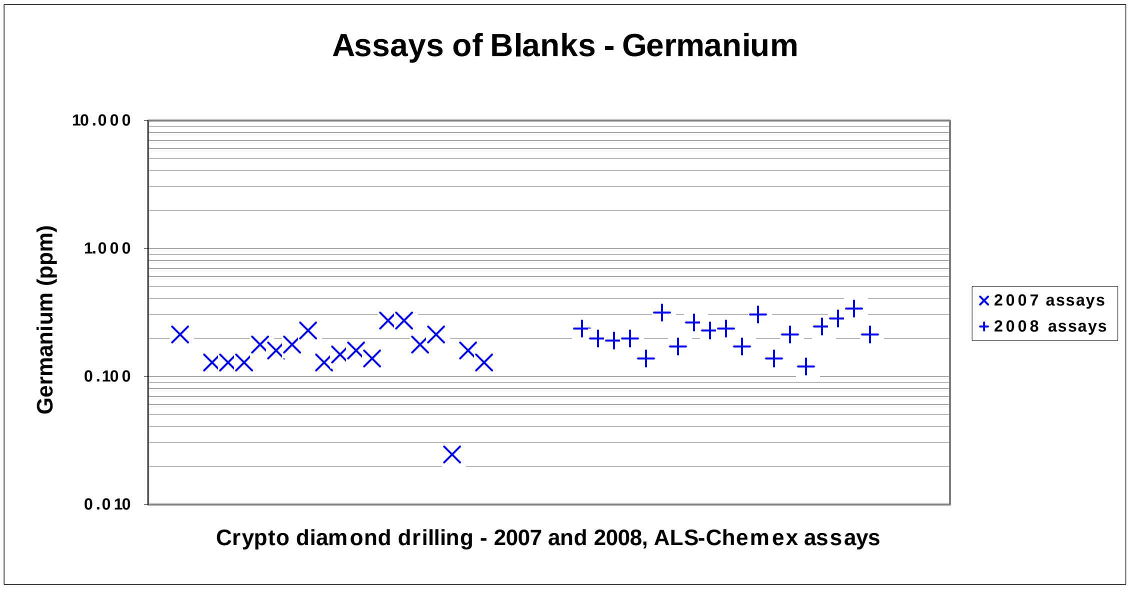


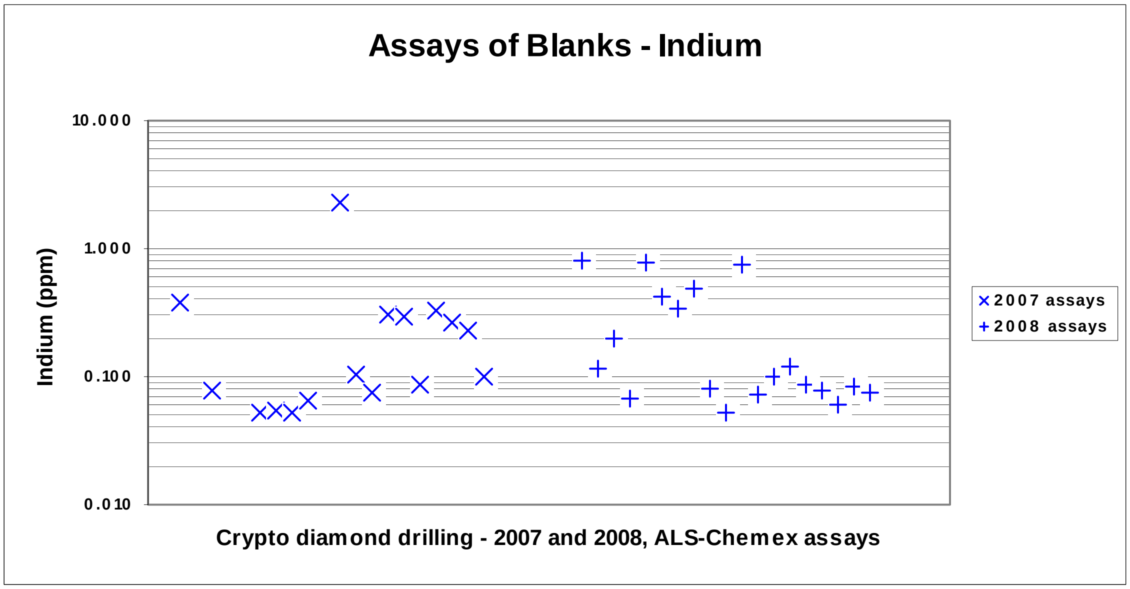
Assays of Blanks - Zinc
1.000
0.100
0.010
0.001
Zinc (%)
Crypto diamond drilling - 2007 and 2008, ALS-Chemex assays
2007 assays
2008 assays


Assays of Blanks - Molybdenum
100.0
10.0
1.0
Molybdenum (ppm)
2007 assays
2008 assays


Assays of Blanks - Cadmium
10.000
1.000
0.100
0.010
Cadmium (ppm)
Crypto diamond drilling - 2007 and 2008, ALS-Chemex assays
2007 assays
2008 assays


Assays of Blanks - Gallium
100.0
10.0
1.0
Gallium (ppm)
Crypto diamond drilling - 2007 and 2008, ALS-Chemex assays
2007 assays
2008 assays


Assays of Blanks - Germanium
10.000
1.000
0.100
0.010
Germanium (ppm)
Crypto diamond drilling - 2007 and 2008, ALS-Chemex assays
2007 assays
2008 assays



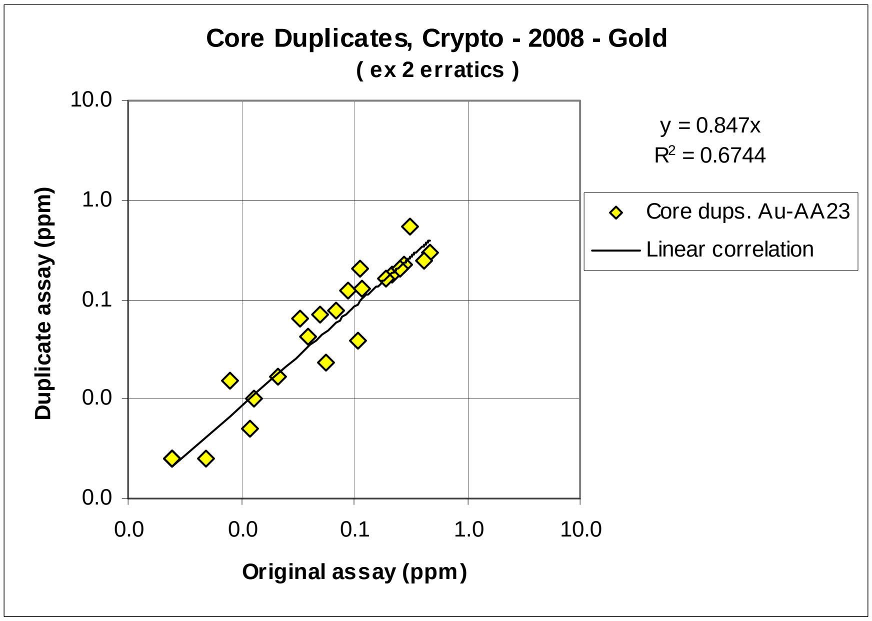
In general, the performance of analysis of these blank samples is not perfect, but is acceptable, especially since the results for the metals of main interest, *viz* zinc, lead, copper and indium, are reasonable.

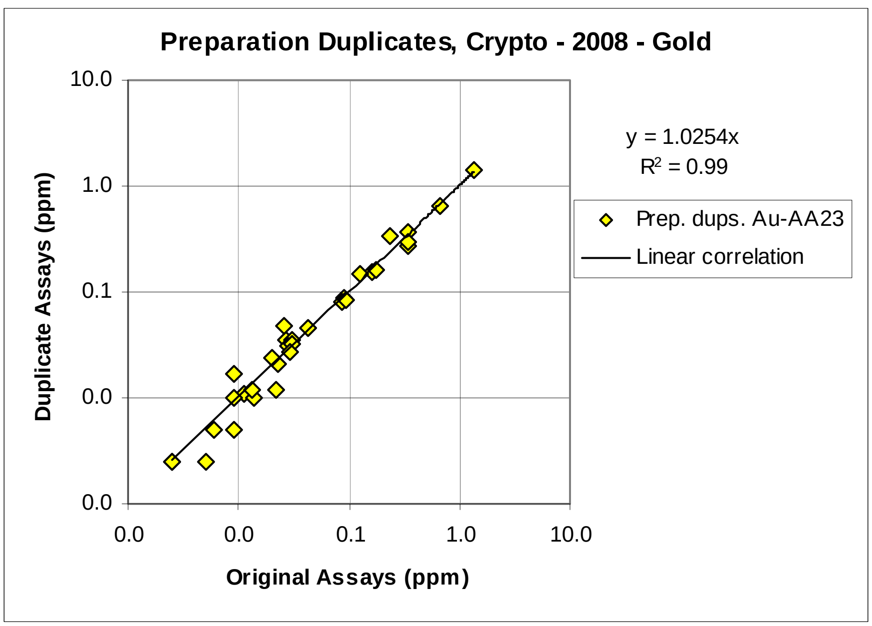
Duplicate assay plots – general statement:

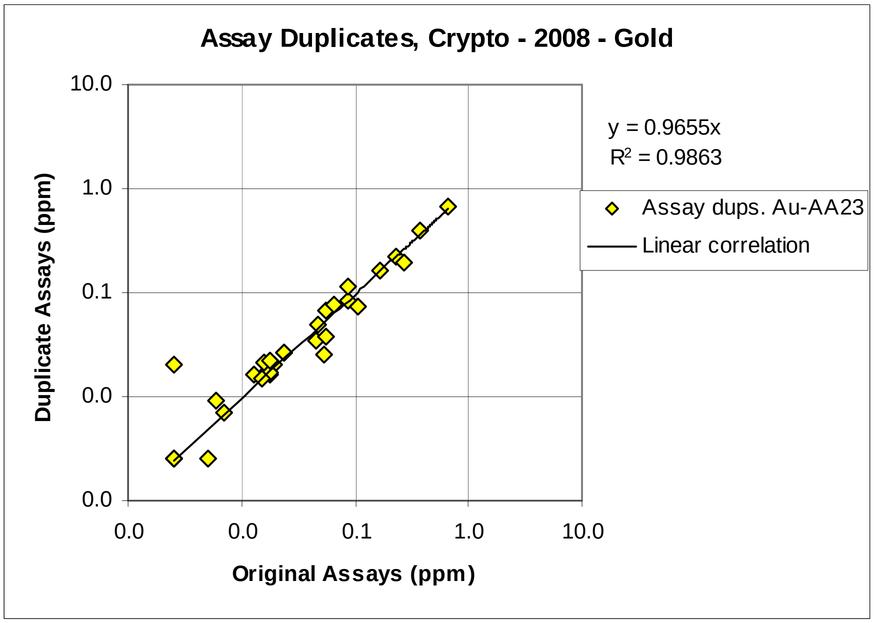
For the 2007 and 2008 diamond drill programs, Lithic inserted: field duplicate samples (sacrificing the second half of core samples); preparation duplicates, where the preparation facility was requested to make duplicate pulps for analysis); and assay duplicates, where the analytical laboratory was requested to make two analyses of a single pulp. These duplicates were meant to be placed at regular intervals in the sample stream, but as selected groups of samples were in some cases separated out for "ore-grade" (OG62) analyses, the order of duplicate sample insertion was not perfect. However, enough duplicate analyses are available to make correlation plots, including analyses by both the MS61 and OG62 methods.

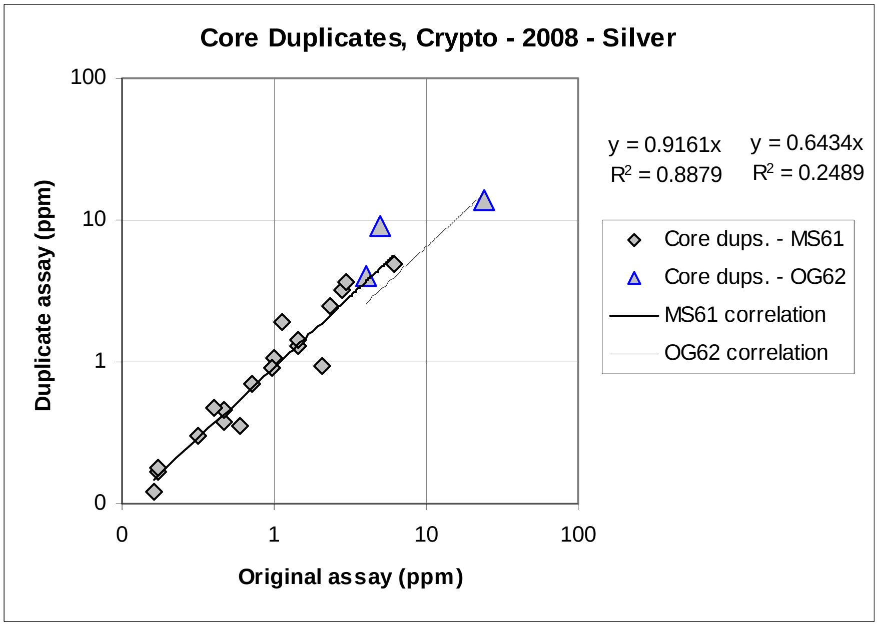
Correlation plots for both drill campaigns follow. Plots are shown separately for all three types of duplicates. The chart titles say 2008, but the data include the 2007 results. In a very few cases, the results suggest sampling mix-ups; these data (totalling three pairs) have been excluded.

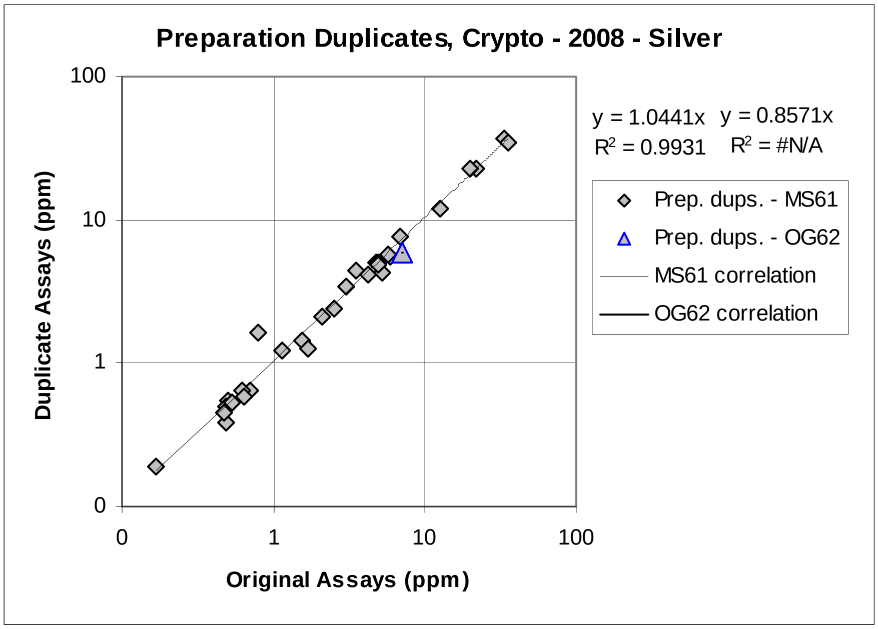
Note the logarithmic scales. Note also that the plots show results for both analytical methods, identified by symbols. In most cases, correlation characteristics are given for data for both methods – the equations to the left are for MS61; to the right for OG62.

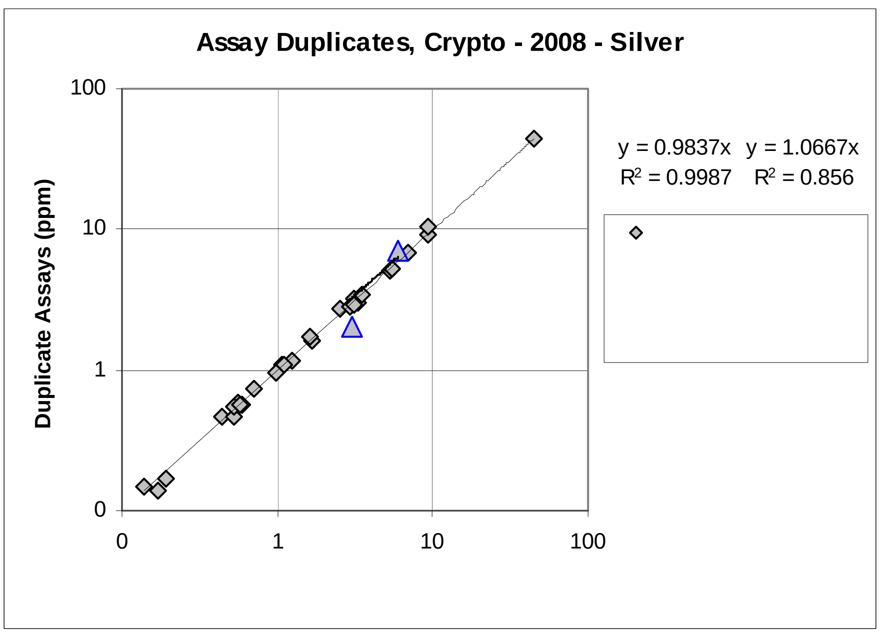

Core Duplicates, Crypto - 2008 - Gold
(ex 2 erratics)
10.0
1.0
0.1
0.0
0.0
Duplicate assay (ppm)
0.0
0.0
0.1
1.0
10.0
Original assay (ppm)
y = 0.847x
R^2 = 0.6744
Core dups. Au-AA23
Linear correlation

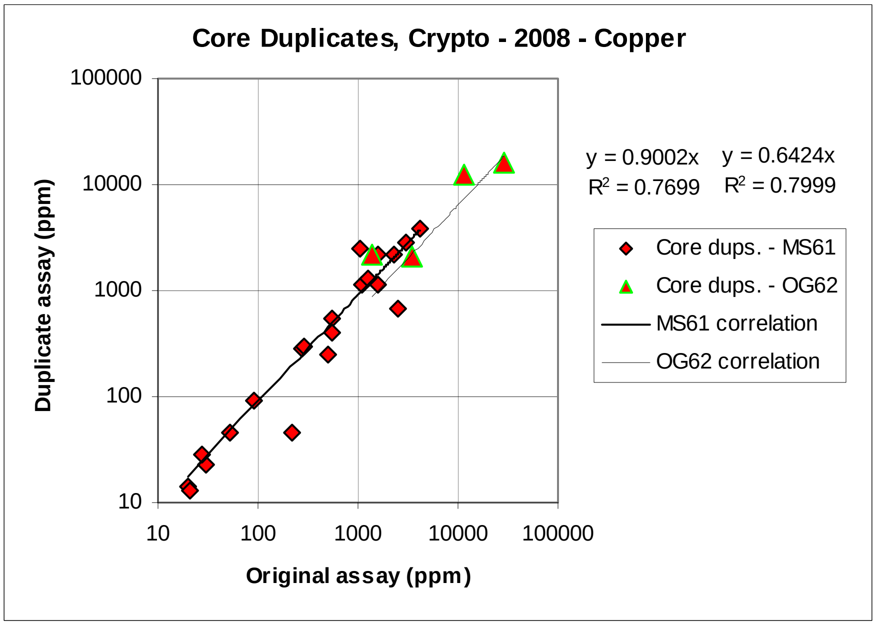

Preparation Duplicates, Crypto - 2008 - Gold
10.0
1.0
0.1
0.0
0.0
Duplicate Assays (ppm)
0.0
0.0
0.1
1.0
10.0
Original Assays (ppm)
y = 1.0254x
R^2 = 0.99
Prep. dups. Au-AA23
Linear correlation

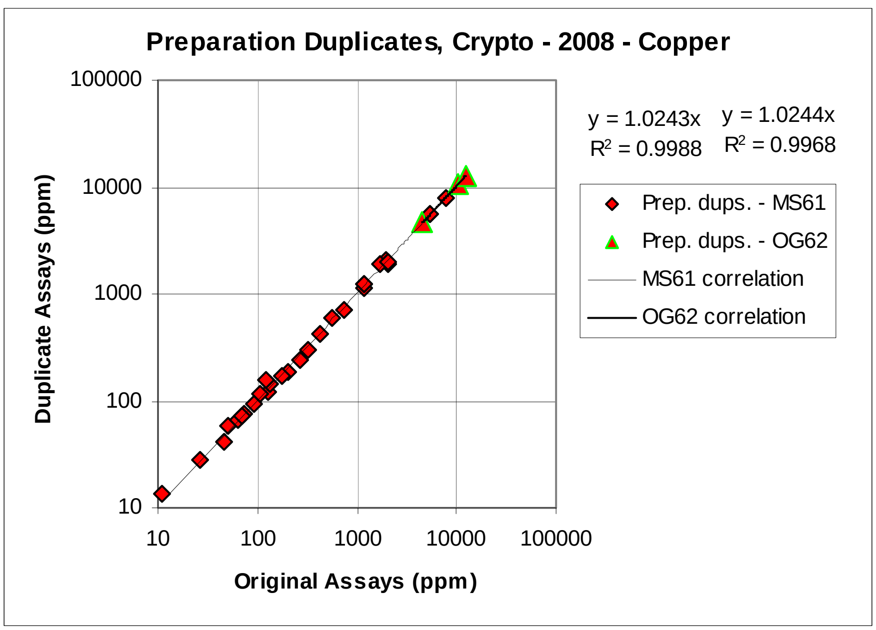

Assay Duplicates, Crypto - 2008 - Gold
10.0
1.0
0.1
0.0
0.0
Duplicate Assays (ppm)
0.0
0.0
0.1
1.0
10.0
Original Assays (ppm)
y = 0.9655x
R2 = 0.9863
Assay dups. Au-AA23
Linear correlation

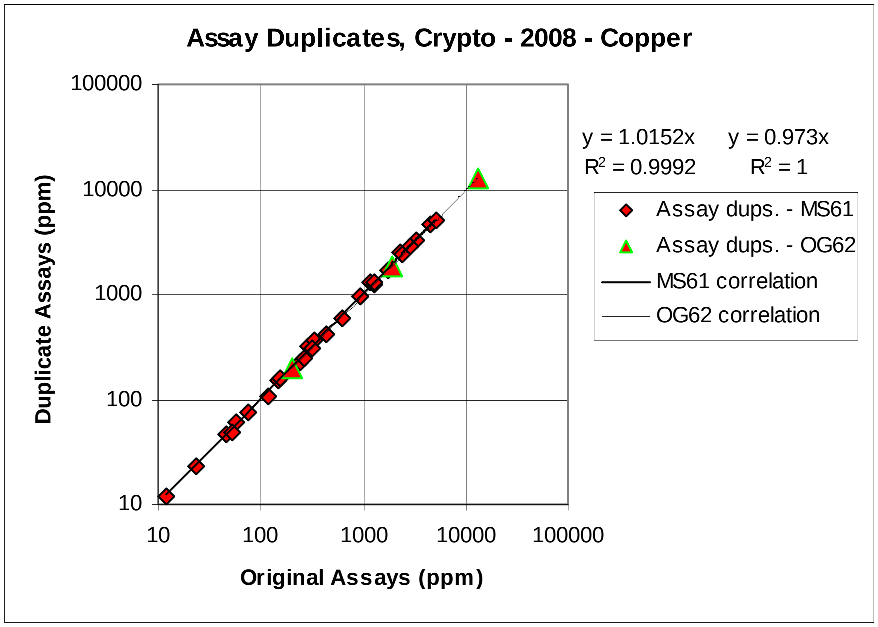

Core Duplicates, Crypto - 2008 - Silver
100
10
1
0
Duplicate assay (ppm)
0
1
10
100
Original assay (ppm)
y = 0.9161x
R2 = 0.8879
y = 0.6434x
R2 = 0.2489
Core dups. - MS61
Core dups. - OG62
MS61 correlation
OG62 correlation

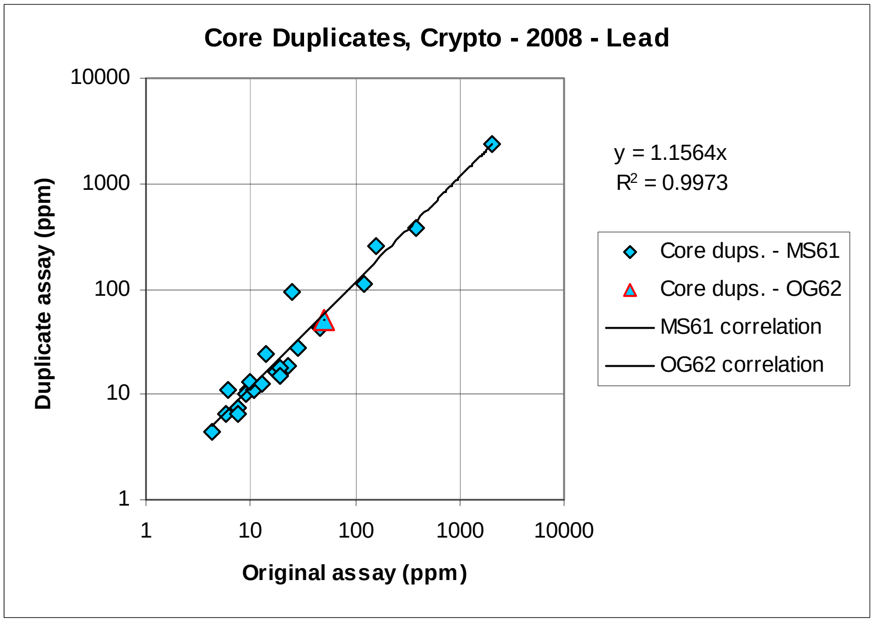

Preparation Duplicates, Crypto - 2008 - Silver
Duplicate Assays (ppm)
Original Assays (ppm)
0
1
10
100
y = 1.0441x
R2 = 0.9931
y = 0.8571x
R2 = #N/A
Prep. dups. - MS61
Prep. dups. - OG62
MS61 correlation
OG62 correlation

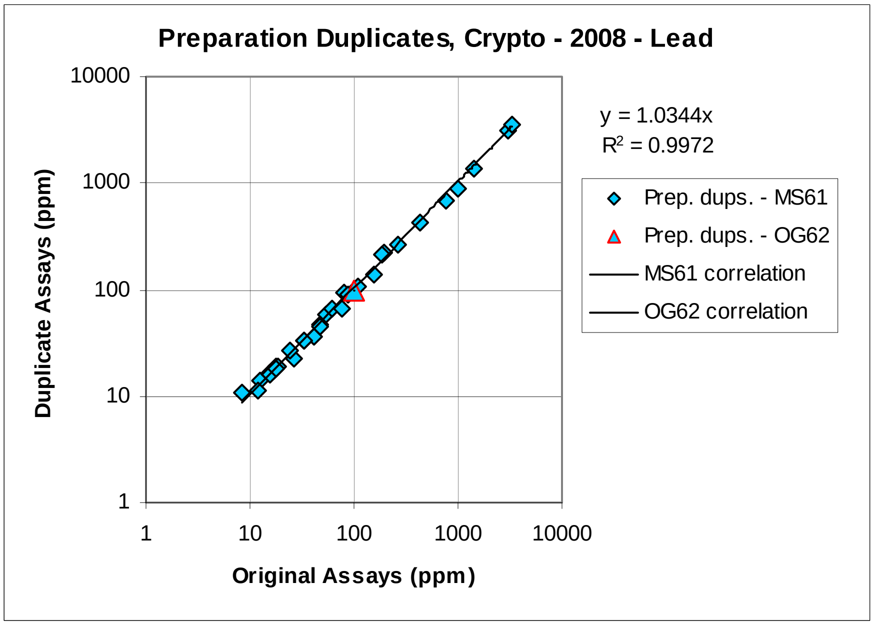

Assay Duplicates, Crypto - 2008 - Silver
Duplicate Assays (ppm)
0
1
10
100
y = 0.9837x
R2 = 0.9987
y = 1.0667x
R2 = 0.856

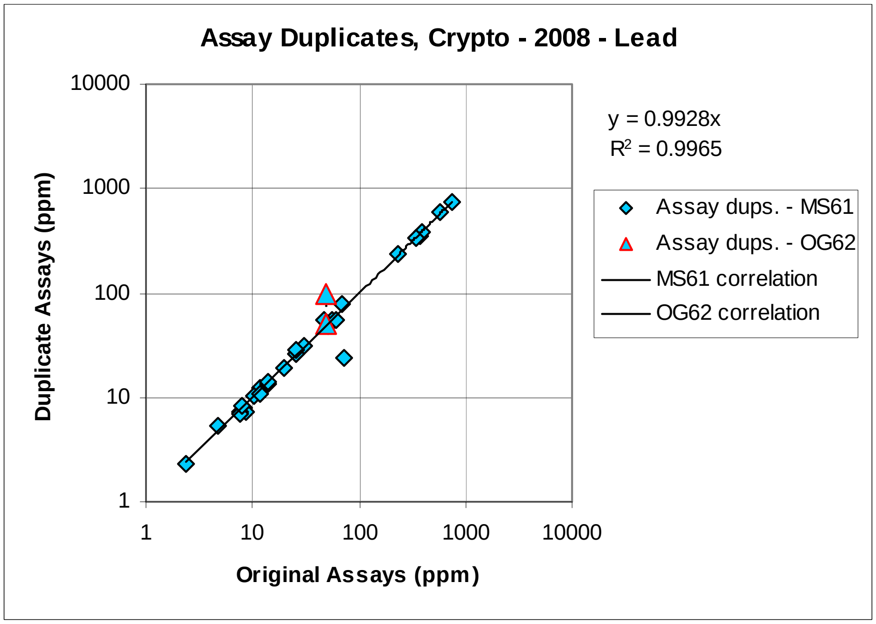

Core Duplicates, Crypto - 2008 - Copper
Duplicate assay (ppm)
Original assay (ppm)
100000
10000
1000
100
10
10
100
1000
10000
100000
y = 0.9002x
R² = 0.7699
y = 0.6424x
R² = 0.7999
Core dups. - MS61
Core dups. - OG62
MS61 correlation
OG62 correlation

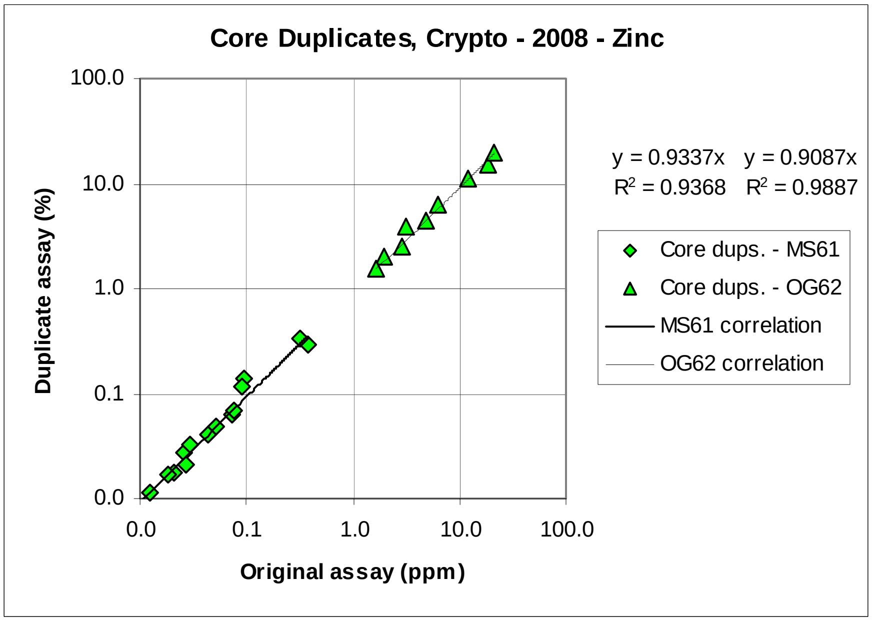

Preparation Duplicates, Crypto - 2008 - Copper
Duplicate Assays (ppm)
Original Assays (ppm)
100000
10000
1000
100
10
10
100
1000
10000
100000
y = 1.0243x
R² = 0.9988
y = 1.0244x
R² = 0.9968
Prep. dups. - MS61
Prep. dups. - OG62
MS61 correlation
OG62 correlation

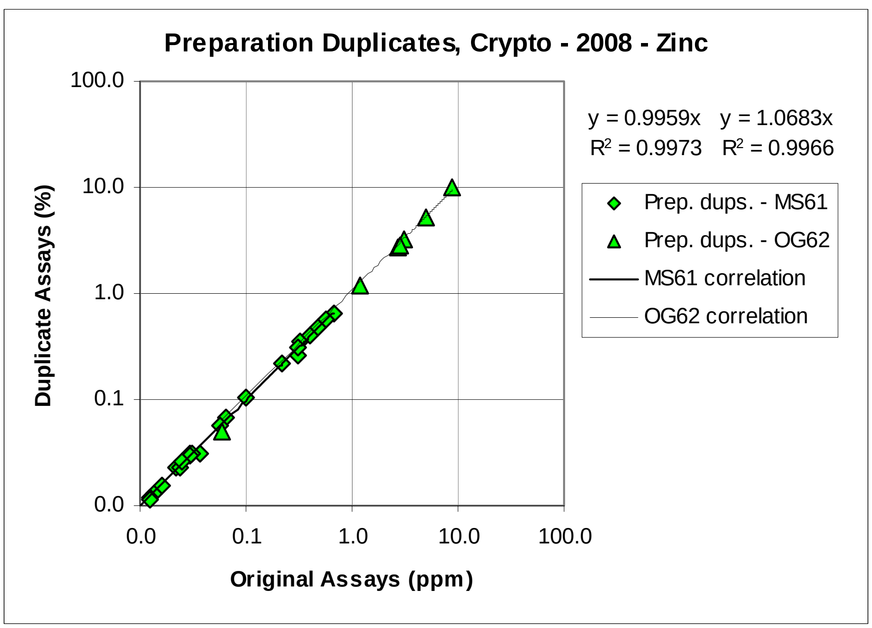

Assay Duplicates, Crypto - 2008 - Copper
Duplicate Assays (ppm)
Original Assays (ppm)
10
100
1000
10000
100000
y = 1.0152x
R^2 = 0.9992
y = 0.973x
R^2 = 1
Assay dups. - MS61
Assay dups. - OG62
MS61 correlation
OG62 correlation

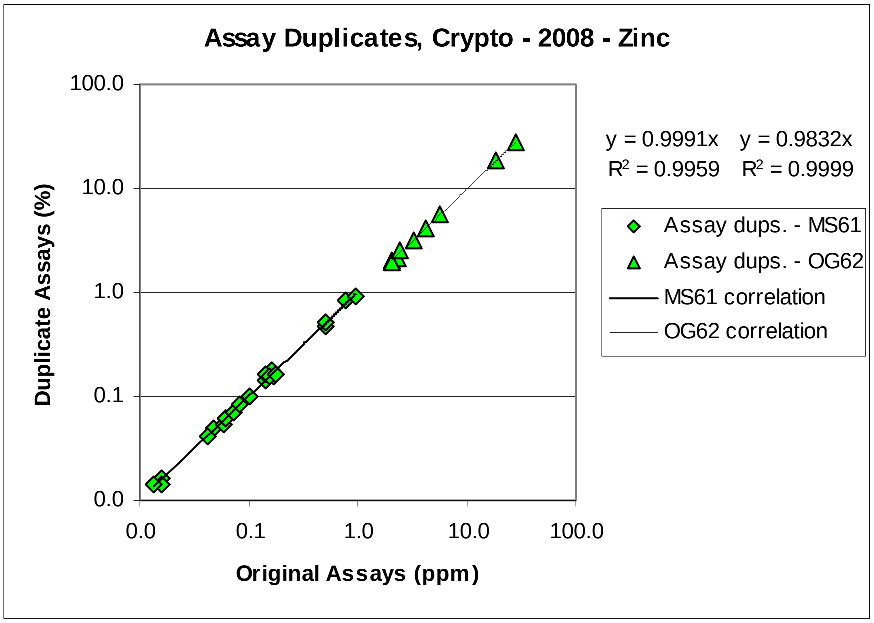

Core Duplicates, Crypto - 2008 - Lead
Duplicate assay (ppm)
Original assay (ppm)
1
10
100
1000
10000
y = 1.1564x
R^2 = 0.9973
Core dups. - MS61
Core dups. - OG62
MS61 correlation
OG62 correlation

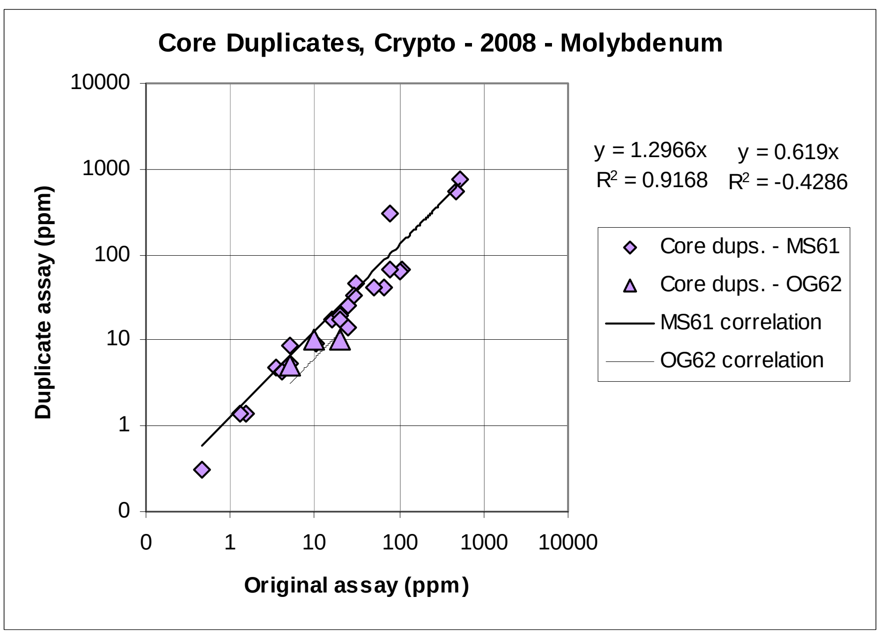

Preparation Duplicates, Crypto - 2008 - Lead
Duplicate Assays (ppm)
Original Assays (ppm)
10000
1000
100
10
1
1
10
100
1000
10000
y = 1.0344x
R^2 = 0.9972
Prep. dups. - MS61
Prep. dups. - OG62
MS61 correlation
OG62 correlation

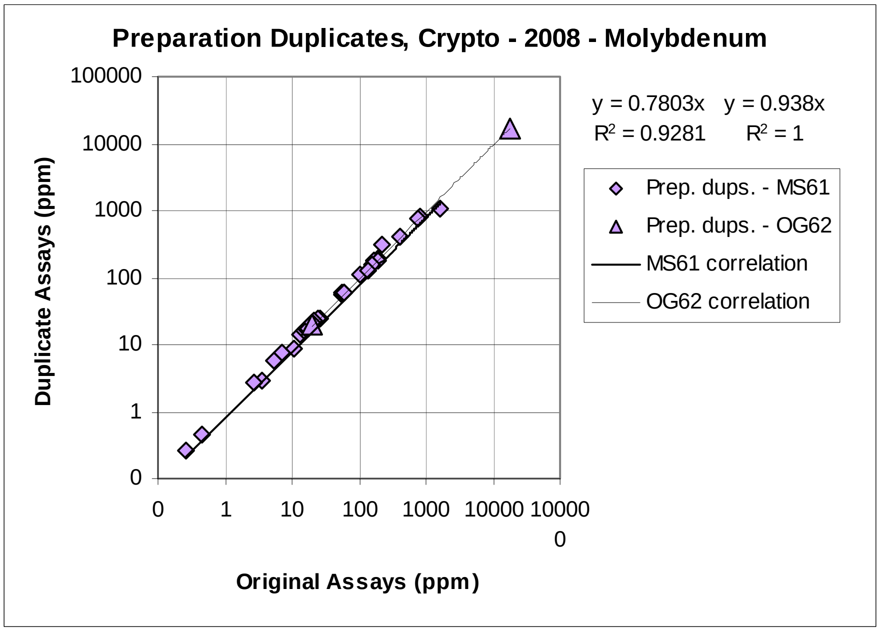

Assay Duplicates, Crypto - 2008 - Lead
Duplicate Assays (ppm)
Original Assays (ppm)
10000
1000
100
10
1
1
10
100
1000
10000
y = 0.9928x
R^2 = 0.9965
Assay dups. - MS61
Assay dups. - OG62
MS61 correlation
OG62 correlation

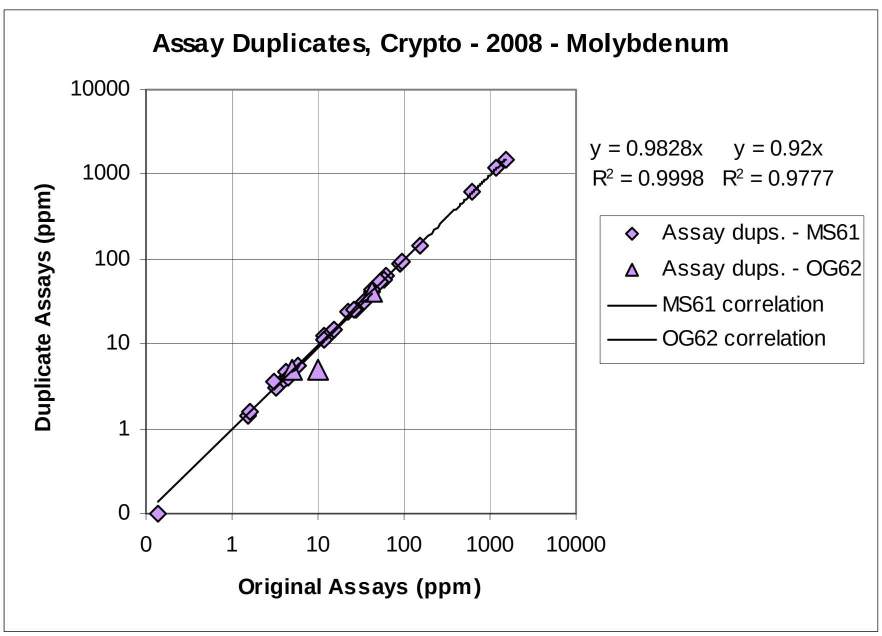

Core Duplicates, Crypto - 2008 - Zinc
100.0
10.0
1.0
0.1
0.0
0.0
0.1
1.0
10.0
100.0
Duplicate assay (%)
Original assay (ppm)
y = 0.9337x y = 0.9087x
R2 = 0.9368 R2 = 0.9887
Core dups. - MS61
Core dups. - OG62
MS61 correlation
OG62 correlation

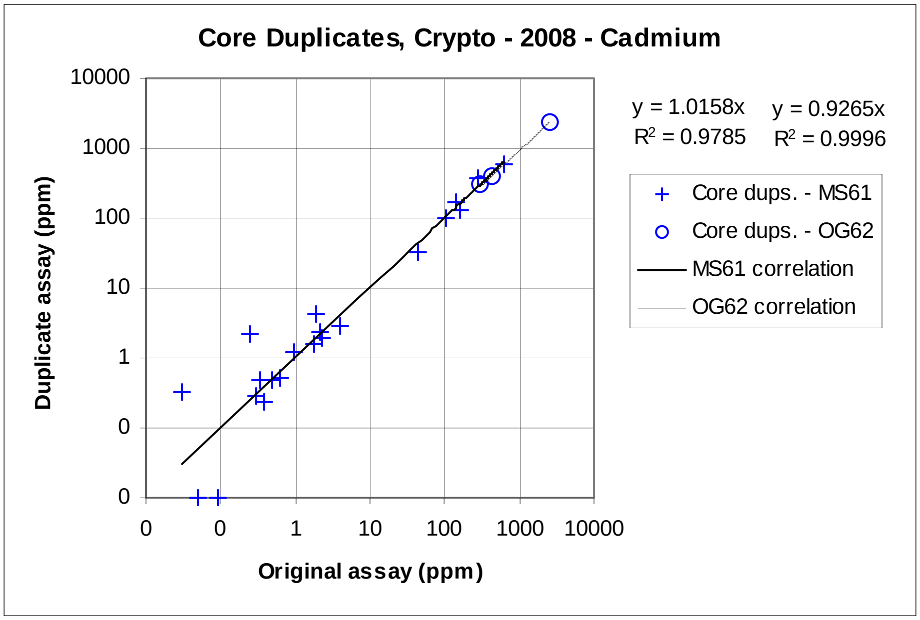

Preparation Duplicates, Crypto - 2008 - Zinc
100.0
10.0
1.0
0.1
0.0
0.0
0.1
1.0
10.0
100.0
Duplicate Assays (%)
Original Assays (ppm)
y = 0.9959x y = 1.0683x
R2 = 0.9973 R2 = 0.9966
Prep. dups. - MS61
Prep. dups. - OG62
MS61 correlation
OG62 correlation

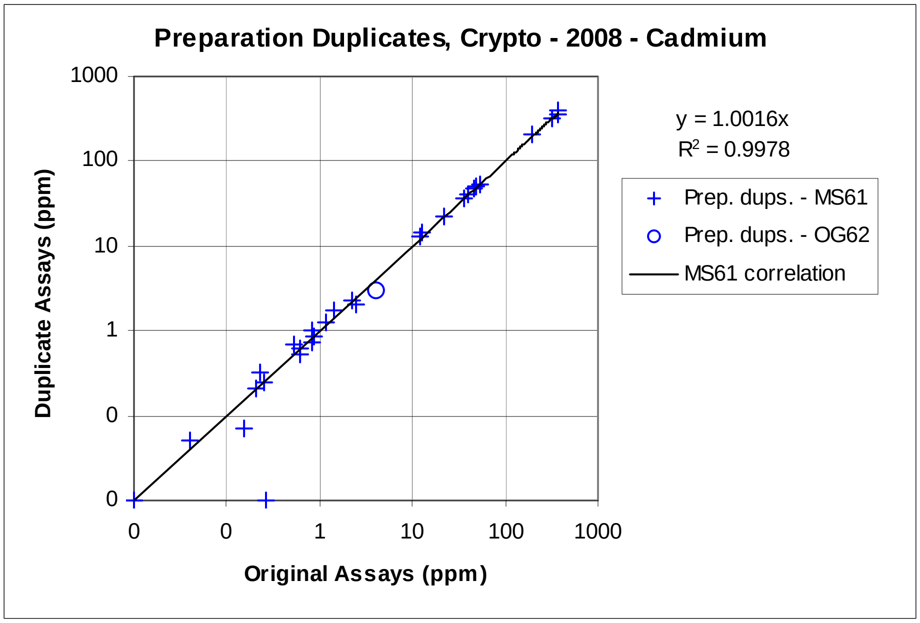

Assay Duplicates, Crypto - 2008 - Zinc
Duplicate Assays (%)
Original Assays (ppm)
100.0
10.0
1.0
0.1
0.0
y = 0.9991x
R² = 0.9959
y = 0.9832x
R² = 0.9999
Assay dups. - MS61
Assay dups. - OG62
MS61 correlation
OG62 correlation

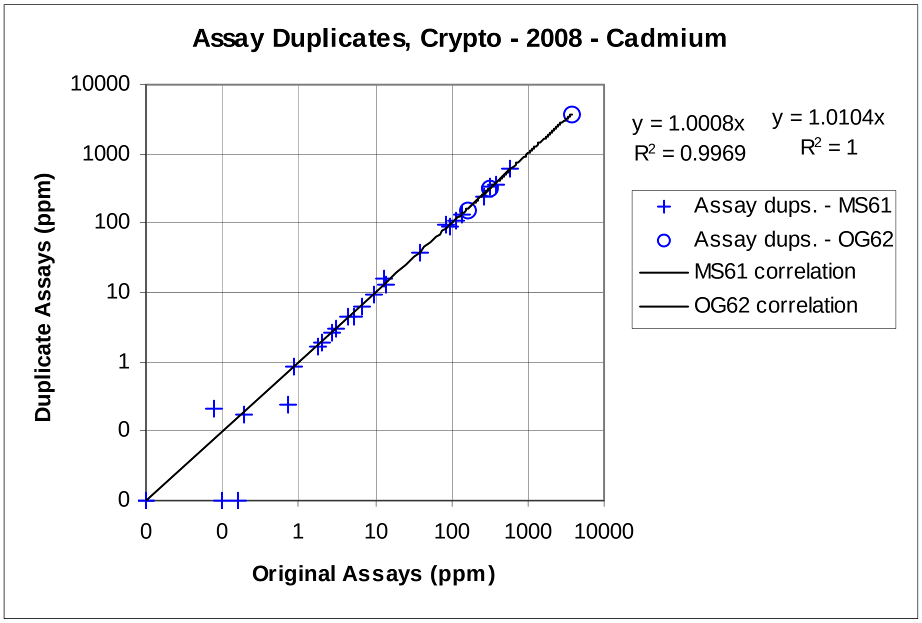

Core Duplicates, Crypto - 2008 - Molybdenum
Duplicate assay (ppm)
Original assay (ppm)
10000
1000
100
10
1
0
y = 1.2966x
R² = 0.9168
y = 0.619x
R² = -0.4286
Core dups. - MS61
Core dups. - OG62
MS61 correlation
OG62 correlation

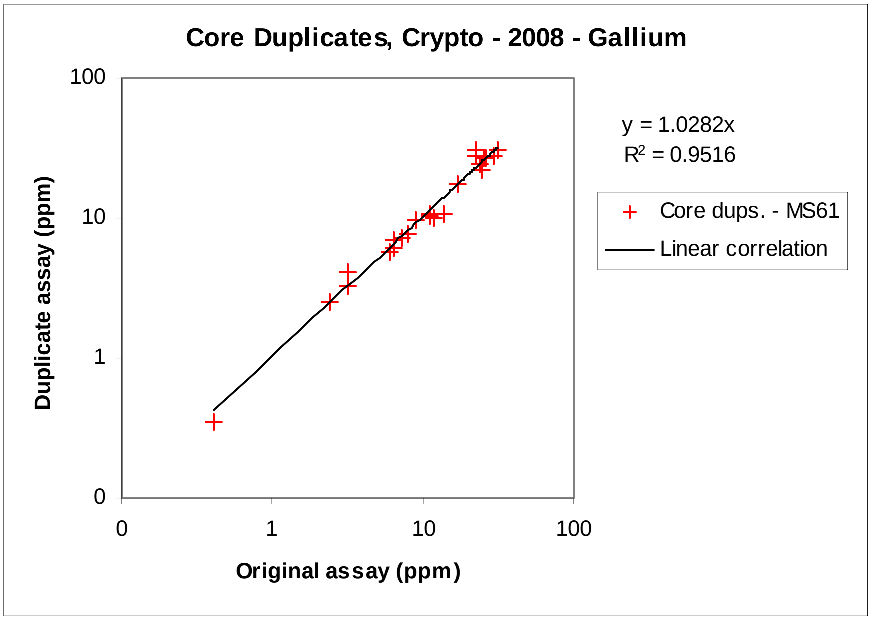

Preparation Duplicates, Crypto - 2008 - Molybdenum
Duplicate Assays (ppm)
Original Assays (ppm)
100000
10000
1000
100
10
1
0
0
1
10
100
1000
10000
100000
y = 0.7803x
R² = 0.9281
y = 0.938x
R² = 1
Prep. dups. - MS61
Prep. dups. - OG62
MS61 correlation
OG62 correlation

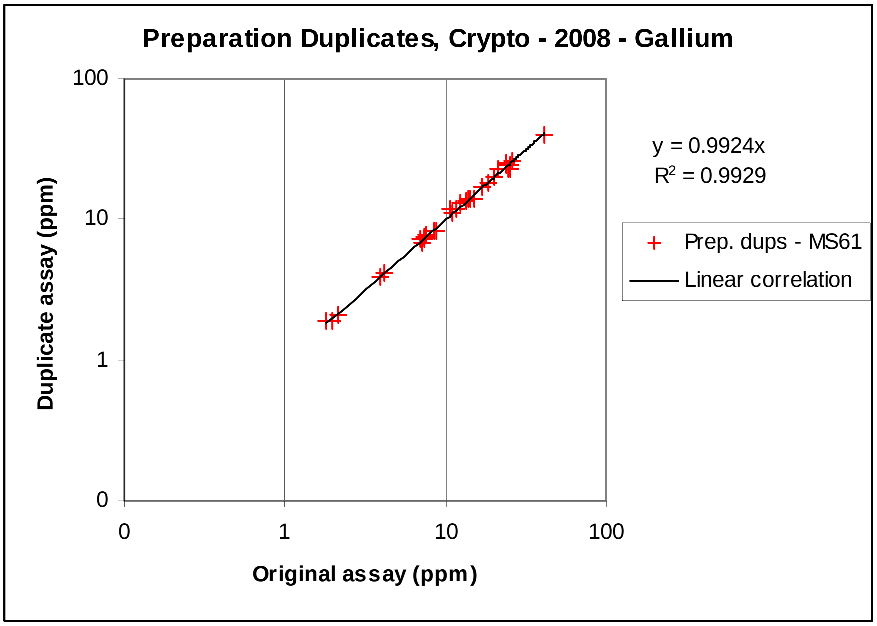

Assay Duplicates, Crypto - 2008 - Molybdenum
Duplicate Assays (ppm)
Original Assays (ppm)
10000
1000
100
10
1
0
0
1
10
100
1000
10000
y = 0.9828x
R² = 0.9998
y = 0.92x
R² = 0.9777
Assay dups. - MS61
Assay dups. - OG62
MS61 correlation
OG62 correlation


Core Duplicates, Crypto - 2008 - Cadmium
Duplicate assay (ppm)
10000
1000
100
10
1
0
0
0
0
1
10
100
1000
10000
Original assay (ppm)
y = 1.0158x
R² = 0.9785
y = 0.9265x
R² = 0.9996
Core dups. - MS61
Core dups. - OG62
MS61 correlation
OG62 correlation


Preparation Duplicates, Crypto - 2008 - Cadmium
Duplicate Assays (ppm)
1000
100
10
1
0
0
0
0
1
10
100
1000
Original Assays (ppm)
y = 1.0016x
R² = 0.9978
Prep. dups. - MS61
Prep. dups. - OG62
MS61 correlation


Assay Duplicates, Crypto - 2008 - Cadmium
Duplicate Assays (ppm)
10000
1000
100
10
1
0
0
0
0
1
10
100
1000
10000
Original Assays (ppm)
y = 1.0008x
R2 = 0.9969
y = 1.0104x
R2 = 1
Assay dups. - MS61
Assay dups. - OG62
MS61 correlation
OG62 correlation


Core Duplicates, Crypto - 2008 - Gallium
Duplicate assay (ppm)
100
10
1
0
0
1
10
100
Original assay (ppm)
y = 1.0282x
R2 = 0.9516
Core dups. - MS61
Linear correlation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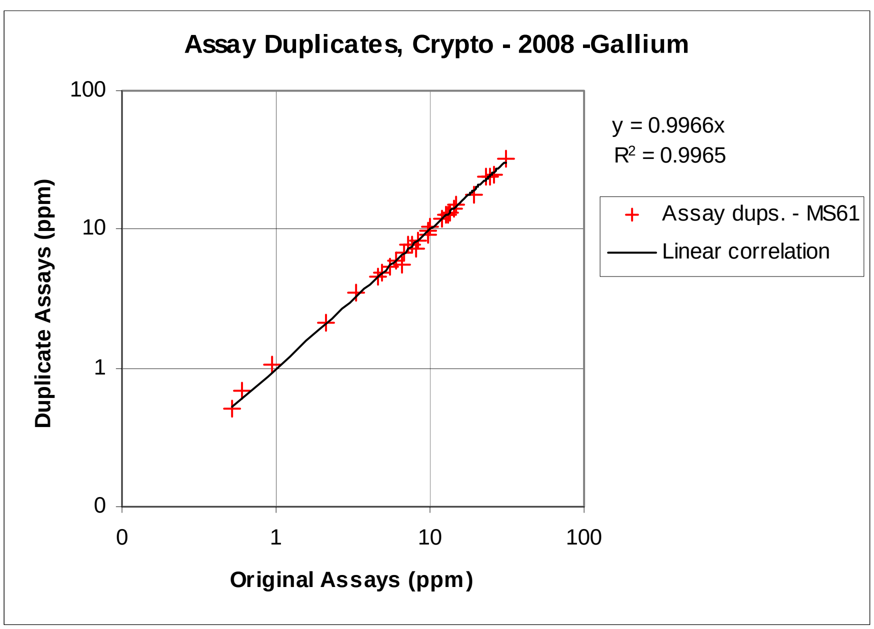
Linear correlation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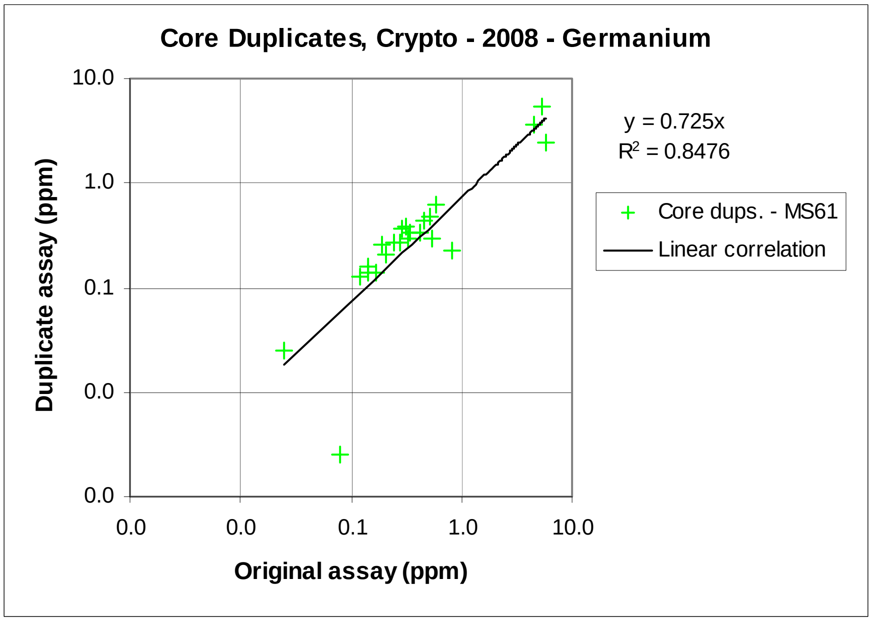
R2 = 0.9965
Assay dups. - MS61
Linear correlation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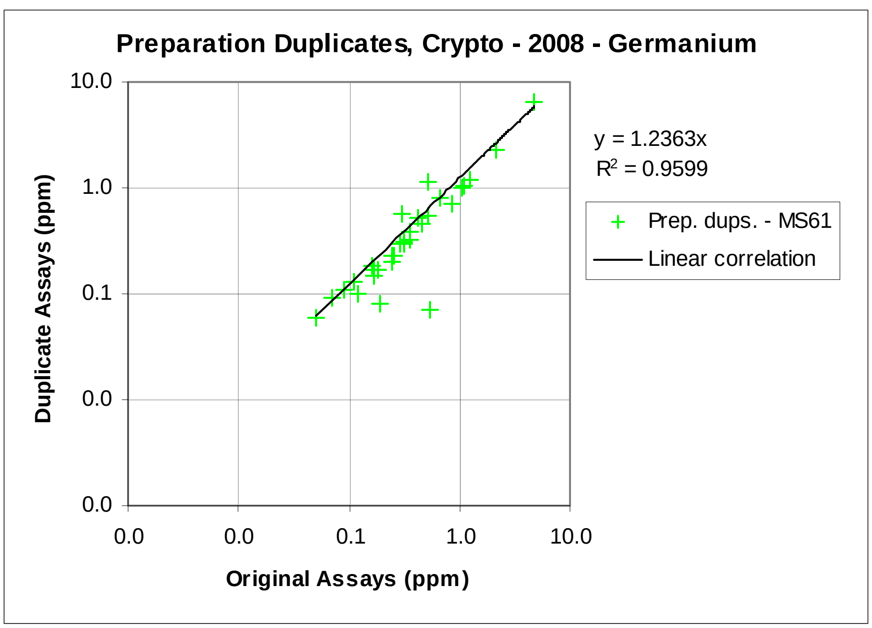
0.1
1.0
10.0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Linear correlation
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Original Assays (ppm)
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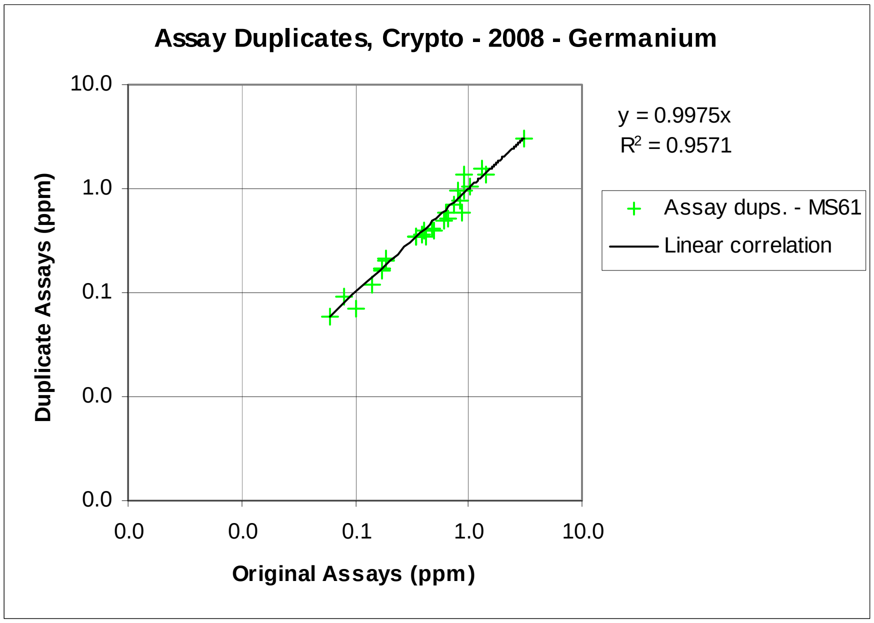
1.0
0.1
0.0
0.0
0.0
0.0
0.1
1.0
10.0


Assay Duplicates, Crypto - 2008 - Germanium
10.0
1.0
0.1
0.0
0.0
Duplicate Assays (ppm)
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0.0
0.1
1.0
10.0
Original Assays (ppm)
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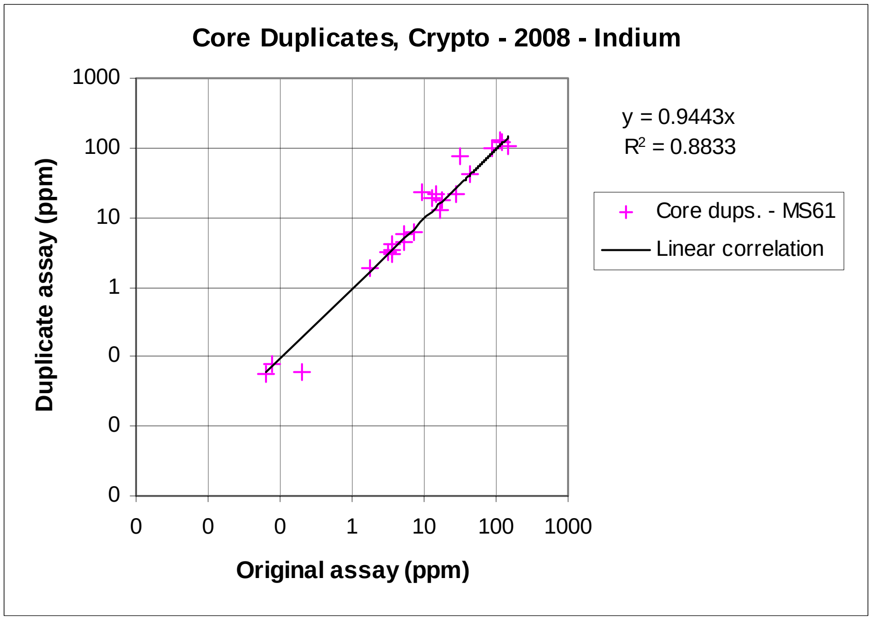
Assay dups. - MS61
Linear correlation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100
10
1
0
0
0
Duplicate assay (ppm)
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0
1
10
100
1000
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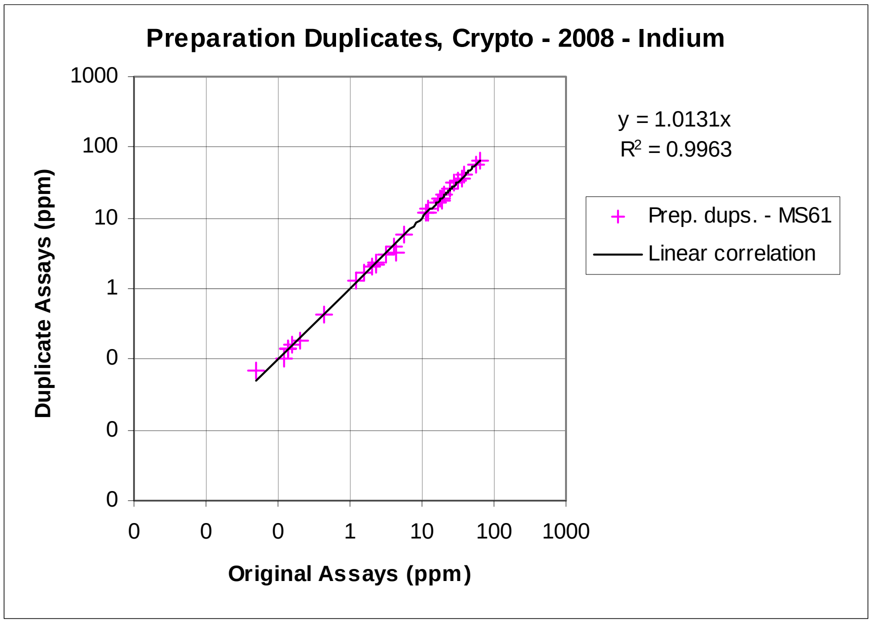
Core dups. - MS61
Linear correlation



Preparation Duplicates, Crypto - 2008 - Indium
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1
0
0
0
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0
0
1
10
100
1000
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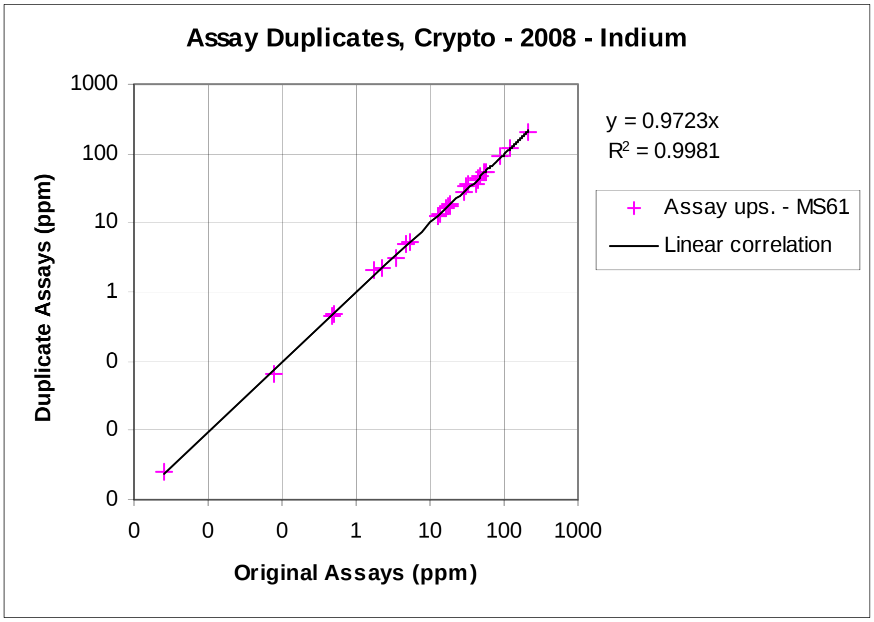
Prep. dups. - MS61
Linear correlation



Assay Duplicates, Crypto - 2008 - Indium
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10
1
0
0
0
Duplicate Assays (ppm)
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0
0
1
10
100
1000
Original Assays (ppm)
y = 0.9723x
R2 = 0.9981
Assay ups. - MS61
Linear correlation

Generally speaking, and especially for the metals of prime interest, the duplicate analyses suggest relatively good precision. As expected, the correlations in most cases are better as one proceeds along the sampling process from core to preparation to pulp re-assay, as one would hope.

Data presented in this review suggest that the various quality control procedures have been successful in showing that assay data from the Crypto diamond drilling in 2007 and 2008 are in general terms acceptable.

Should additional drilling be undertaken, a more rigorous method of inserting control samples should be instituted. In particular, it will be important that larger groups of samples are chosen for individual sample shipments for "ore-grade" assays, and that these shipments have controls inserted in regular rotation. Different standard materials may well be applicable for these shipments. These shipments should be analysed for the entire suite of elements of interest. The main point here is that the insertion of control samples be keyed to sample shipments, rather than being set up beforehand on the basis of drilling intervals.

On the subject of blanks, there are some minor concerns with respect to the levels of a few of the analysed elements in the rhyolite used as a blank. For future drill programs, it would be advisable to look for an alternate material. There is reportedly a massive intrusive body in the general area; it would be a valuable exercise to collect some of this material and have a small number (say ten or so) of samples analysed for the entire suite of elements of interest.

"G. R. Peatfield"

Giles R. Peatfield, Ph.D., P.Eng.

Appendix C

Descriptive Statistics of Metal Domain Samples

Zn Oxide Zone 101 **Capping None**

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	302					15.24	211.62	m
To	302					17.07	212.20	m
Length	302	1.81	1.57			0.15	6.09	m
Zn	302	0.200	0.267	0.283	1.062	0.001	2.500	%
Zn_cap	302	0.200	0.267	0.283	1.062	0.001	2.500	%
Zn_dmn	302					100	100	

Zn Oxide Zone 102 **Capping None**

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	291					20.09	208.48	m
To	291					21.47	210.01	m
Length	291	1.53	1.45			0.15	8.23	m
Zn_1	291	1.500	2.241	2.061	0.919	0.001	15.600	%
Zn_cap	291	1.500	2.241	2.061	0.919	0.001	15.600	%
Zn_dmn	291					200	200	

Zn Oxide Zone 103 **Capping None**

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	83					17.07	200.44	m
To	83					18.90	201.63	m
Length	83	1.33	1.19			0.15	4.05	m
Zn_1	83	13.800	16.317	8.677	0.532	1.400	43.390	%
Zn_cap	83	13.800	16.317	8.677	0.532	1.400	43.390	%
Zn_dmn	83					300	300	

Zn Sulfide Zone 101 **Capping 13 % Zn**

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	1,543					147.37	885.14	m
To	1,543					149.59	887.88	m
Length	1,543	1.54	1.52			0.09	7.01	m
Zn	1,543	0.340	0.736	1.093	1.485	0.001	18.600	%
Zn_cap	1,543	0.340	0.735	1.079	1.468	0.001	13.000	%
Zn_dmn	1,543					100	100	

Zn Sulfide Zone 102 **Capping 18 % Zn**

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	535					156.90	747.98	m
To	535					159.80	751.51	m
Length	535	1.45	1.50			0.13	4.15	m
Zn_23	535	4.680	5.047	3.177	0.629	0.034	30.050	%
Zn_cap	535	4.680	5.015	2.981	0.595	0.034	18.000	%
Zn_dmn	535					200	200	

Zn Sulfide Zone 103 **Capping None**

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	185					182.58	682.75	m
To	185					184.56	684.73	m
Length	185	1.32	1.35			0.06	3.20	m
Zn_23	185	15.450	16.568	7.314	0.441	0.267	46.500	%
Zn_cap	185	15.450	16.568	7.314	0.441	0.267	46.500	%
Zn_dmn	185					300	300	

Cu Oxide Zone 201 Capping 0.6 % Cu

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	296					17.07	211.62	m
To	296					18.90	212.20	m
Length	296	1.48	1.37			0.15	8.23	m
Cu	296	0.081	0.109	0.108	0.999	0.000	1.295	%
Cu_cap	296	0.081	0.107	0.094	0.886	0.000	0.600	%
Cu_dmn	296					100	100	

Cu Oxide Zone 202 Capping 1 % Cu

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	42					25.30	169.77	m
To	42					26.21	172.67	m
Length	42	1.25	1.07			0.45	3.05	m
Cu_1	42	0.410	0.471	0.263	0.559	0.098	2.100	%
Cu_cap	42	0.410	0.454	0.188	0.415	0.098	1.000	%
Cu_dmn	42					200	200	

Cu Oxide Zone 203 Capping None

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	18					41.15	197.82	m
To	18					42.67	199.95	m
Length	18	1.07	1.11			0.30	2.13	m
Cu_1	18	1.442	2.605	2.340	0.898	0.713	7.840	%
Cu_cap	18	1.442	2.605	2.340	0.898	0.713	7.840	%
Cu_dmn	18					300	300	

Cu Sulfide Zone 201 Capping 1.5 % Cu

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	1,173					156.67	885.14	m
To	1,173					158.19	887.88	m
Length	1,173	1.48	1.52			0.11	7.01	m
Cu_23	1,173	0.141	0.170	0.154	0.906	0.001	3.300	%
Cu_cap	1,173	0.141	0.169	0.136	0.804	0.001	1.500	%
Cu_dmn	1,173					100	100	

Cu Sulfide Zone 202 Capping 2.5 % Cu

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	506					172.91	878.46	m
To	506					173.96	882.79	m
Length	506	1.42	1.40			0.09	4.88	m
Cu_23	506	0.437	0.490	0.295	0.602	0.003	3.570	%
Cu_cap	506	0.437	0.490	0.291	0.595	0.003	2.500	%
Cu_dmn	506					200	200	

Cu Sulfide Zone 203 Capping 7 % Cu

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	199					203.45	749.81	m
To	199					203.91	751.33	m
Length	199	1.29	1.22			0.06	3.29	m
Cu_23	199	1.095	1.392	1.101	0.791	0.022	14.380	%
Cu_cap	199	1.095	1.378	0.976	0.708	0.022	7.000	%
Cu_dmn	199					300	300	

In Oxide Zone 301 Capping None

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	104					20.09	175.26	m
To	104					21.47	176.33	m
Length	104	1.45	1.50			0.30	3.68	m
In	104	2.05	3.01	3.43	1.14	0.04	20.40	g /t
In_cap	104	2.05	3.01	3.43	1.14	0.04	20.40	g /t
In_dmn	104					100	100	

In Area 1 Zone 302 303 Capping 90 ppm In

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	58					53.10	177.85	m
To	58					53.60	178.77	m
Length	58	1.12	1.06			0.20	2.68	m
In_1	58	18.05	28.42	28.10	0.99	5.41	149.50	g /t
In_cap	58	18.05	26.56	21.10	0.79	5.41	90.00	g /t
In_dmn	58					200	300	

In Sulfide Zone 301 Capping None

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	371					149.96	763.83	m
To	371					150.72	765.35	m
Length	371	1.38	1.46			0.11	3.05	m
In	371	4.09	4.67	3.34	0.72	0.16	28.70	g /t
In_cap	371	4.09	4.67	3.34	0.72	0.16	28.70	g /t
In_dmn	371					100	100	

In Sulfide Zone 302 Capping 250 ppm In

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	1,015					156.90	774.80	m
To	1,015					159.80	776.48	m
Length	1,015	1.44	1.50			0.15	4.15	m
In_23	1,015	22.60	31.24	28.71	0.92	0.19	385.00	ppm
In_cap	1,015	22.60	31.09	27.11	0.87	0.19	250.00	ppm
In_dmn	1,015					200	200	

In Sulfide Zone 303 Capping 500 ppm In

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
From	125					245.52	731.52	m
To	125					246.74	733.50	m
Length	125	1.41	1.50			0.30	2.59	m
In_23	125	137.50	159.20	115.18	0.72	24.50	1055.00	ppm
In_cap	125	137.50	154.08	85.78	0.56	24.50	500.00	ppm
In_dmn	125					300	300	

Appendix D

Letter Report from Knight Piesold Consulting Regarding Conceptual Tailings Facility

Knight Piésold
CONSULTING

File No.: VA101-428/1-A.00
Cont. No.:VA10-00898

Suite 1400 - 750 West Pender Street
Vancouver, BC Canada V6C 2T8

Tel: 604.685.0543
Fax: 604.685.0147
www.knightpiesold.com

June 1, 2010

Mr. Chris Staargaard
President & CEO
Lithic Resources Ltd.
912 - 510 West Hastings Street
Vancouver, BC V6B 1L8

Dear Chris,

Re: Desktop Conceptual Engineering Study for the Crypto-Zinc-Copper-Indium Project

Knight Piésold (KPL) has completed a desktop conceptual engineering study for the Crypto Zinc Copper Project in Utah, USA. The desktop study included preliminary layouts for a Tailings Storage Facility (TSF), surface waste rock dump, and associated water management diversions to support a Preliminary Economic Assessment (PEA) study.

Introduction

The Crypto-Zinc-Copper-Indium Project is a proposed mine in Western Utah. The deposit is located at the south end of the Salt Flats, approximately 100 miles southwest of Salt Lake City. The proposed mining method will be longitudinal retreat stoping at a rate of 3,500 tpd (1.26 million tpy). The total milled is approximately 12.1 million tonnes over a ten year mine life. Lithic Resources is currently completing a PEA for the project and contracted Knight Piésold to develop preliminary layouts and costs for the TSF to support a PEA. This letter presents the preliminary layout and cost estimate for the TSF.

Site Characteristics

The project site is located on the western side of a small mountain range on the south end of the Salt Flats. The site ranges in elevation from approximately 1,320 m along the flats to approximately 1,800 m. The deposit is located at an approximate elevation of 1,350 m. The project site is located in an arid environment with minimal precipitation and high evaporation.

TSF Design Concept

The objective of the TSF design concept was to locate a TSF as close a possible to the deposit while minimizing embankment fill requirements. It was also a requirement to maintain a 150 m buffer zone from the Pony Express Road. The TSF was thus located on the gentle sloping (1 to 2% grade) ground immediately west of the deposit between the Pony Express Road and the main road to the west of the deposit. The preliminary layout of the TSF included a large impoundment footprint, as the volumetric storage efficiency (volume of storage vs. volume of embankment fill materials) for TSFs on relatively flat terrain increases as the TSF footprint increases. The embankment fill materials (and costs) are comparatively lower than for facilities with smaller footprints and higher embankments.

The preliminary layout for the TSF was to provide containment for 12.1 million tonnes of tailings solids at an assumed dry density of 1.4 tonnes/m^3. The preliminary design basis for the TSF is shown on Table 1.

The TSF has been located on the flats to the west of the deposit where the elevation ranges from approximately 1,324 m to 1336 m. The ultimate elevation of the TSF embankments is approximately 1,340 m and has embankments that range in height from 16 m at the western corner of the facility to approximately 4 m along the eastern side of the TSF. The final arrangement for the TSF is shown on Figure 1.

The TSF covers an approximate area of 1.1 Mm^2 and includes a low permeability basin liner/subgrade material with a basin underdrain for seepage control. The embankments were assumed to be constructed using the downstream construction method with a 3H:1V upstream slope to facilitate placement and compaction of the low permeability basin liner, and a 2H:1V downstream slope. The embankments are homogeneous dams with appropriate filter zones to prevent internal erosion of the embankment fill materials in the event the liner system if damaged. The embankments were assumed to be constructed using local borrow materials.

The low permeability basin liner/subgrade material was assumed to be appropriate for seepage control at this stage. This should be revisited once the tailings geochemistry is better understood.

The tailings were assumed to be conventional slurry tailings with a solids content of 30%, as per discussions with Lithic resources. The tailings were discharged from the embankment crest around the impoundment to maximize the tailings distribution within the TSF to improve the storage efficiency. A small supernatant pond was assumed to be located on the western side of the facility for the PEA.

The starter dam sized for containment of two years of operations at the full throughput. A mill ramp-up period was not considered for the PEA.

Water Management

The project site is located in an arid environment and there will be little water available to the process from local surface runoff. The water table is apparently relatively close to surface (approximately 1,300 m elevation) however, it is not known if a sustainable well field can be developed to provide the required water for the process, estimated at approximately 1,500 gpm for a throughput of 3,500 gpm at 30% solids. An allowance has been included in the cost estimate for make-up water which may involve sourcing water from across the valley to the west.

The make-up water requirements for the mill can be significantly reduced by thickening the tailings at the mill and capturing the thickener overflow rather than discharging the water to the TSF where it evaporates. Thickening tailings is usually cost effective in arid environments where water is in short supply. There are other potential options for optimizing the pipeworks and deposition systems to accommodate thickened tailings and these should be considered as potential opportunities to consider in future studies.

The management of runoff from storm events was not considered as part of the PEA. The layout of the TSF included a freeboard allowance of 2 m at all times, however, the adequacy of this assumption will be reviewed in future studies once the site hydrometeorology is better understood. The ultimate footprint of the TSF has embankments around the entire facility, which reduces the runoff from storm events to that generated from direct precipitation on the TSF. Prior to this, diversion ditches will be constructed to divert runoff from the upstream catchment around the TSF.

Closure and Reclamation

The TSF will be required to maintain long-term physical and geochemical stability, protect the downstream environment, and manage surface water. The preliminary closure and reclamation plan for the TSF includes constructing a TSF cover at closure and construction of an emergency overflow spillway.

Cost Estimate

The cost estimate for the PEA has been divided in five components, as follows

1. Tailings Storage Facility, including Site Preparation and TSF construction
2. Electrical
3. Mechanical, including tailings and reclaim pipeworks and pumps
4. Instrumentation, and
5. Closure and Reclamation.

The annual cost summary, including capital and operating costs are shown on Table 2. The initial and sustaining capex cost summary is shown on Table 3. The opex summary costs are shown on Table 4. The itemized annual capital costs are shown on Table 5. The capital cost estimate includes allowances for mobilization and demobilization (8% of direct costs), indirects (20% of direct costs), and a contingency of 25% for additional cost items not identified during the PEA. The itemized opex costs are shown on Table 6. The opex cost estimate also includes contingency allowance of 25%. A power cost of 5.5 cents per kWh was used to develop the opex costs.

The total capital cost over the mine life is approximately $47 million, with a total opex cost of approximately $4.6 million. The initial capital cost is approximately $19 million. The cost estimate for the PEA is a high level cost estimate and there may be opportunities to optimize the design concept once the site geotechnical conditions and tailings geochemistry are better understood. Additionally, the use of thickened tailings should be evaluated in future studies as this is typically cost effective in arid climates.

We trust the attached meets your requirements for a PEA. Please do not hesitate to contact the undersigned should you have any questions.

Yours truly,
KNIGHT PIESOLD LTD.

PROFESSIONAL PROVINCE OF L.J. GALBRAITH 25493 BRITISH COLUMBIA ENGINEER

Signed:	Approved:
Les Galbraith, P.Eng.	Ken Brouwer, P.Eng.
Senior Engineer	Managing Director

Attachments:

Table 1 Rev 0	Preliminary Design Basis and Operating Criteria
Table 2 Rev 0	Cost Summary
Table 3 Rev 0	Capex Cost Summary - Initial and Sustaining Capital

Table 4 Rev 0	Opex Cost Summary
Table 5 Rev 0	Itemized Capital Costs
Table 6 Rev 0	Itemized Operating Costs
Figure 1 Rev 0	Preliminary Environment Assessment Preliminary Layout of Tailings Storage Facility

/lg

Knight Piésold
CONSULTING

TABLE 1

LITHIC RESOURCES LTD.
CRYPTO ZINC COPPER PROJECT

TAILINGS STORAGE FACILITY
PRELIMINARY DESIGN BASIS AND OPERATING CRITERIA

6/1/2010 13:56

ITEM		DESIGN CRITERIA
1.0 GENERAL DESIGN CRITERIA		
1.1	Mine Production	• Total tons mined – 12.1 million tonnes (Mt) • Throughput – 3,500 tpd (1.26 Mtpy) • Total waste rock – 1.595 Mt • Duration of milling – 10 years
1.2	Tailings Characteristics	• Slurry Solids Content: 30% solids by weight • Tailings Specific Gravity: 2.6 • Settled Dry Density: 1.4 t/m3 • Non-Acid forming tailings
1.3	Site Elevation	• Approximately 1322 m to 1800m.
2.0 TAILINGS STORAGE FACILITY (TSF)		
2.1	Storage Capacity	• 12.1 million tonnes milled – approximately 8.7 million m^3 • Starter dam capacity 2.5 mt (2 years of storage)
2.2	Location Selection	• TSF located a minimum 150 m from the Pony Express Road
2.3	Freeboard	• 2 m freeboard during operations
2.4	Embankment Crest Width	• 10 m minimum
2.5	Supernatant Pond	• Maximum pond volume of 600,000 m^3
2.6	Embankment Slopes	• Downstream fill slope: 2H:1V • Upstream fill slope: 3H:1V
2.7	Embankment Construction Method	• Downstream Construction
2.8	Seepage Control	• Geomembrane covering
3.0 TAILINGS FACILITY PIPEWORKS		
3.1	Bulk Tailings	• Design flow 3,500 TPD at 30% solids • Discharge from crest of TSF embankment
3.2	Reclaim System	• One floating barge pump-station at 8,000 m^3/day
4.0 MAKE-UP WATER		
4.1	Make-up water requirements	• Mill required approximately 8,000 m^3/day for process at 30% solids content
4.2	Reclaim System	• One floating barge pump-station at 8,000 m^3/day
5.0 RECLAMATION AND CLOSURE		
		• Tailings beach not flooded will be covered with a rock cap and re-vegetated • PMF spillway will be installed for emergency

M:\1\01\00428\01\A\Correspondence\VA10-00898\Tables\[Table 1 Rev 01.xls]Table 1

0	21MAY'10	ISSUED WITH LETTER VA10-00898	LJG	BB	KJB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

Knight Piésold CONSULTING

TABLE 2

LITHIC RESOURCES LTD.
CRYPTO ZINC COPPER PROJECT

TAILINGS STORAGE FACILITY
COST SUMMARY

6/1/2010 13:58

Year	TAILINGS STORAGE FACILITY		
	Capital Cost	Operating Cost	Total Cost
-1	$16,200,000	$0	$16,200,000
1	$0	$460,000	$460,000
2	$4,800,000	$460,000	$5,260,000
3	$20,000	$460,000	$480,000
4	$5,100,000	$460,000	$5,560,000
5	$20,000	$460,000	$480,000
6	$5,100,000	$460,000	$5,560,000
7	$20,000	$460,000	$480,000
8	$5,100,000	$460,000	$5,560,000
9	$20,000	$460,000	$480,000
10	$20,000	$460,000	$480,000
Closure	$7,700,000	$0	$7,700,000
Totals	$44,100,000	$4,600,000	$48,700,000

M:\1\01\00428\01\A\Data\[PEA Cost estimate May 25.xlsx]Table 2

0	21MAY'10	ISSUED WITH LETTER VA10-00898	MS	LJG	BB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

Knight Piésold CONSULTING

TABLE 3

LITHIC RESOURCES LTD.
CRYPTO ZINC COPPER PROJECT

TAILINGS STORAGE FACILITY
CAPEX COST SUMMARY - INITIAL AND SUSTAINING CAPITAL

Print Feb/24/09 14:55:25

WORK ACTIVITY	TAILINGS STORAGE FACILITY CAPITAL COST BREAKDOWN
	Total Cost
Site Preparation	$950,000
Tailings Facility	$19,960,000
Electrical	$150,000
Mechanical (Tailings and reclaim piping and pumpstations)	$2,640,000
Instrumentation	$200,000
Closure and Reclamation	$5,000,000
Sub Total	$28,900,000
Mob/Demob (8%)	$2,310,000
Indirects (20%)	$5,780,000
Contingency (25%)	$7,220,000
TOTAL	$44,210,000

M:\1\01\00428\01\A\Data\[PEA Cost estimate May 25.xlsx]Table 3

0	21MAY'10	ISSUED WITH LETTER VA-10-00898	MS	LJG	BB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

Knight Piésold CONSULTING

TABLE 4

LITHIC RESOURCES LTD.
CRYPTO ZINC COPPER PROJECT

TAILINGS STORAGE FACILITY
OPEX COST SUMMARY

Print Feb/24/09 14:55:25

WORK ACTIVITY	TAILINGS STORAGE FACILITY OPERATING COST BREAKDOWN Total Cost
Service Road Maintenance	$500,000
Maintenance and Manpower	$1,000,000
Power - Tailings, Reclaim Water and Seepage Recycle	$60,000
Environmental Compliance	$500,000
Engineering Support and Reporting	$500,000
Sub Total	$2,560,000
Contingency (25%)	$920,000
TOTAL	$3,480,000

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0	21MAY'10	ISSUED WITH LETTER VA10-00898	MS	LJG	BB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D

Knight Piésold
CONSULTING

TABLE 5

LITHIC RESOURCES LTD.
CRYPTO ZINC COPPER PROJECT

TAILINGS STORAGE FACILITY
ITEMIZED CAPITAL COSTS

Print Feb/24/09 14:55:25

Item Number	Description	Units	Unit Cost[1]	Year -1 Cost	Year 1 Cost	Year 2 Cost	Year 3 Cost	Year 4 Cost	Year 5 Cost	Year 6 Cost	Year 7 Cost	Year 8 Cost	Year 9 Cost	Year 10 Cost	Year 11 Cost	Total
1	**TAILINGS STORAGE FACILITY**															
A	**Site Preparation**															
	Clearing, grubbing, topsoil removal	m^3	$2.50	$750,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$750,000
	Service road	km	$10,000	$50,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$50,000
	Haul road construction	km	$25,000	$50,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$50,000
	Sediment and erosion control BMP's allowance	LS	$100,000	$100,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$100,000
			Sub-Total	**$950,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$950,000**
B	**Tailings Facility**															
	Dam footprint foundation preparation	m^2	$0.40	$54,000	$0	$20,000	$0	$20,000	$0	$20,000	$0	$20,000	$0	$0	$0	$134,000
	Embankment Filter/Transition Zone	m^3	$12.00	$720,000	$0	$360,000	$0	$360,000	$0	$360,000	$0	$360,000	$0	$0	$0	$2,160,000
	Low Permeability Basin Liner/Subgrade	m^3	$6.00	$2,237,400	$0	$0	$0	$125,334	$0	$125,334	$0	$125,334	$0	$0	$0	$2,613,402
	Basin Underdrain System	m^3	$5.00	$559,350	$0	$0	$0	$37,600	$0	$37,600	$0	$37,600	$0	$0	$0	$672,151
	Embankment Shell Zone	m^3	$5.00	$3,150,000	$0	$2,760,000	$0	$2,790,000	$0	$2,790,000	$0	$2,790,000	$0	$0	$0	$14,280,000
	Seepage recovery systems	LS	$100,000	$100,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$100,000
			Sub-Total	**$6,820,750**	**$0**	**$3,140,000**	**$0**	**$3,332,934**	**$0**	**$3,332,934**	**$0**	**$3,332,934**	**$0**	**$0**	**$0**	**$19,959,553**
2	**ELECTRICAL**															
	Powerline	km	$15,000	$45,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$45,000
	E - House with PLC & MCC	ea	$100,000	$100,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$100,000
			Sub-Total	**$145,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$145,000**
3	**MECHANICAL**															
	Tailings pumpstation	ea	$350,000	$350,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$350,000
	Reclaim barge	ea	$250,000	$250,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$250,000
	Tailings Pipeline	m	$100	$520,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$520,000
	Tailings Off-take Valves	ea	$10,000	$300,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$300,000
	Reclaim Water Pipeline	m	$100	$220,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$220,000
	Water Supply Allowance	LS	$1,000,000	$1,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0		$1,000,000
			Sub-Total	**$2,640,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$2,640,000**
4	**INSTRUMENTATION**															
	Geotechnical instrumentation	ea	$3,000	$60,000	$0	$15,000	$15,000	$15,000	$15,000	$15,000	$15,000	$15,000	$15,000	$15,000	$0	$195,000
			Sub-Total	**$60,000**	**$0**	**$15,000**	**$15,000**	**$15,000**	**$15,000**	**$15,000**	**$15,000**	**$15,000**	**$15,000**	**$15,000**	**$0**	**$195,000**
5	**CLOSURE AND RECLAMATION**															
	TSF Cover	m^3	$10	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$5,000,000	
			Sub-Total	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$5,000,000	**$5,000,000**
	SUBTOTAL ITEMS 1.1 TO 1.5			$10,615,750	$0	$3,155,000	$15,000	$3,347,934	$15,000	$3,347,934	$15,000	$3,347,934	$15,000	$15,000	$5,000,000	$28,889,553
	MOBILIZATION/DEMOBILIZATION		8%	$849,260	$0	$252,400	$1,200	$267,835	$1,200	$267,835	$1,200	$267,835	$1,200	$1,200	$400,000	$2,311,164
	INDIRECTS		20%	$2,123,150	$0	$631,000	$3,000	$669,587	$3,000	$669,587	$3,000	$669,587	$3,000	$3,000	$1,000,000	$5,777,911
	CONTINGENCY		25%	$2,653,938	$0	$788,750	$3,750	$836,984	$3,750	$836,984	$3,750	$836,984	$3,750	$3,750	$1,250,000	$7,222,388
	OVERALL PROJECT TOTAL			**$16,242,098**	**$0**	**$4,827,150**	**$22,950**	**$5,122,339**	**$22,950**	**$5,122,339**	**$22,950**	**$5,122,339**	**$22,950**	**$22,950**	**$7,650,000**	**$44,201,015**

M:\1\01\00428\01\A\Data\[PEA Cost estimate May 25.xlsx]Table 5

REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D
0	21MAY'10	ISSUED WITH LETTER VA10-00898	MS	LJG	BB

Knight Piésold CONSULTING

TABLE 6

LITHIC RESOURCES LTD.
CRYPTO ZINC COPPER PROJECT

TAILINGS STORAGE FACILITY
ITEMIZED OPERATING COSTS

6/1/2010 14:01

	Description	Units	Unit Cost	Year 1		Year 2		Year 3		Year 4		Year 5		Year 6		Year 7		Year 8		Year 9		Year 10	
				Quantity	Cost	Quantity	Cost	Quantity	Cost	Quantity	Cost	Quantity	Cost	Quantity	Cost	Quantity	Cost	Quantity	Cost	Quantity	Cost	Quantity	Cost
	ANNUAL OPERATING COST																						
	Service road maintenance	km	$10,000	5	$50,000	5	$50,000	5	$50,000	5	$50,000	5	$50,000	5	$50,000	5	$50,000	5	$50,000	5	$50,000	5	$50,000
	Maintenance	year	$20,000	1	$20,000	1	$20,000	1	$20,000	1	$20,000	1	$20,000	1	$20,000	1	$20,000	1	$20,000	1	$20,000	1	$20,000
	Manpower	year	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000
TSF	Tailings and Reclaim Pipeling Maintenance and Replacement (15% of Capital/year)	year	$740,000	1	$111,000	1	$111,000	1	$111,000	1	$111,000	1	$111,000	1	$111,000	1	$111,000	1	$111,000	1	$111,000	1	$111,000
	Power - Tailings pumping	MWh	$5.50	475	$2,613	489	$2,689	503	$2,765	517	$2,842	531	$2,918	544	$2,994	558	$3,071	572	$3,147	586	$3,224	600	$3,300
	Power - Reclaim pumping	MWh	$5.50	400	$2,200	394	$2,169	389	$2,139	383	$2,108	378	$2,078	372	$2,047	367	$2,017	361	$1,986	356	$1,956	350	$1,925
	Power -Seepage recovery pumping	MWh	$5.50	100	$550	106	522	111	544	117	567	122	589	128	$703	133	$733	139	$764	144	$794	150	$825
	Environmental compliance	PS	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000
	Engineering support and reporting	PS	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000
			Sub-Total		$366,363		$366,381		$366,449		$366,517		$366,585		$366,744		$366,821		$366,897		$366,974		$367,050
	CONTINGENCY		25%		$91,591		$91,595		$91,612		$91,629		$91,646		$91,686		$91,705		$91,724		$91,743		$91,763
	OVERALL TOTAL				$457,953		$457,976		$458,061		$458,146		$458,231		$458,431		$458,526		$458,622		$458,717		$458,813

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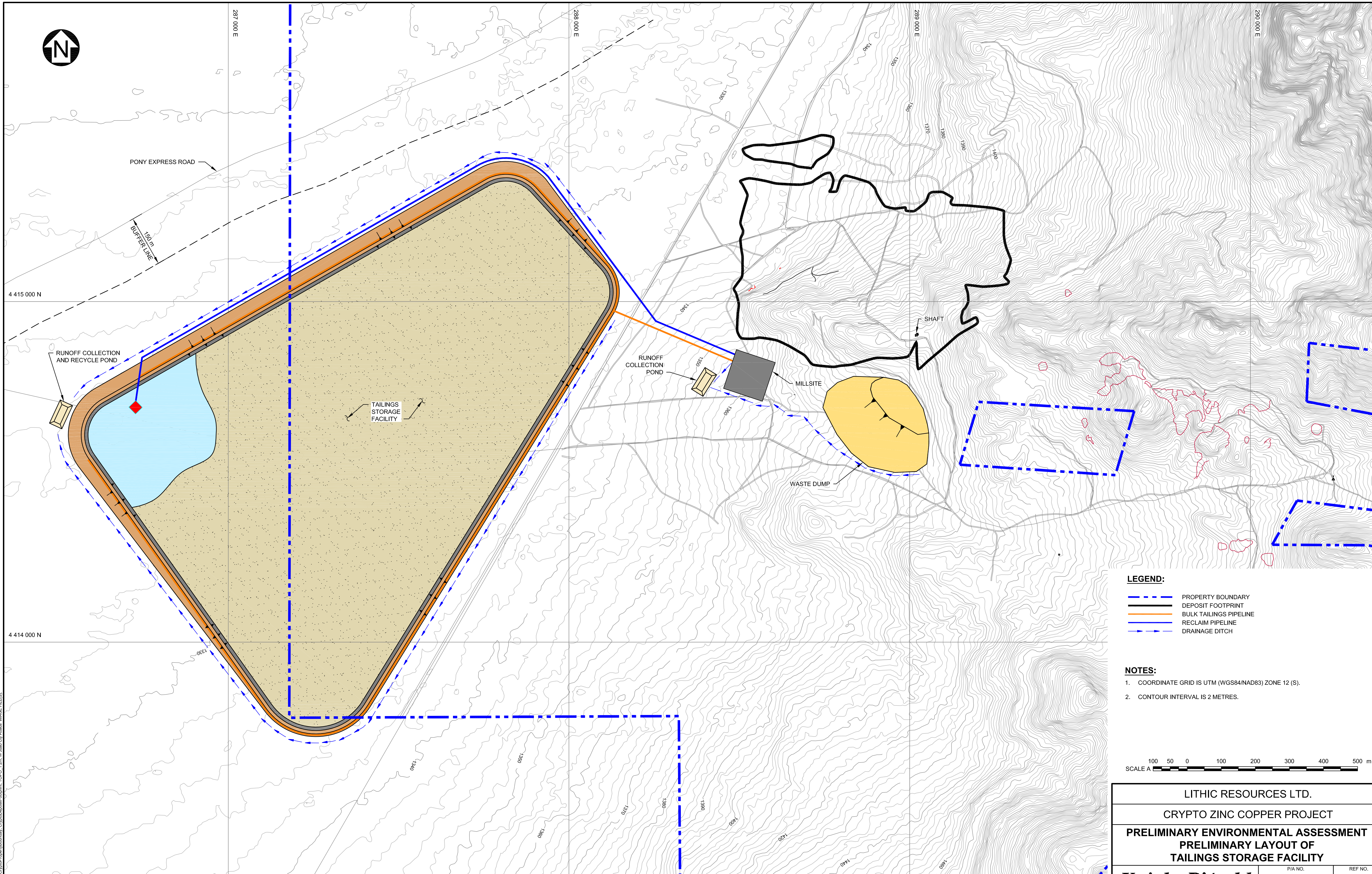
0	21MAY10	ISSUED WITH LETTER VA10-00898	MS	LJG	BB
REV	DATE	DESCRIPTION	PREP'D	CHK'D	APP'D



LEGEND:

- PROPERTY BOUNDARY
- DEPOSIT FOOTPRINT
- BULK TAILINGS PIPELINE
- RECLAIM PIPELINE
- DRAINAGE DITCH

NOTES:

1. COORDINATE GRID IS UTM (WGS84/NAD83) ZONE 12 (S).
2. CONTOUR INTERVAL IS 2 METRES.

SCALE A: 100 50 0 100 200 300 400 500 m

LITHIC RESOURCES LTD.

CRYPTO ZINC COPPER PROJECT

PRELIMINARY ENVIRONMENTAL ASSESSMENT PRELIMINARY LAYOUT OF TAILINGS STORAGE FACILITY

Knight Piésold CONSULTING

P/A NO. VA101-428/1 | REF NO. VA10-00898

FIGURE 1 REV 0

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	21MAY'10	ISSUED WITH LETTER VA10-00898	LJG	TAM	BB	KJB

Appendix E

Cash Flow Model

Cash Flow Model			Sunk Costs		Development Period		Production Period											Totals or Averages
Crypto Project - U/G Mining Operation		**Variables**	**Year -4**	**Year -3**	**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	
Metal Prices																		
Zinc Price	US$/lb	$1.10					$1.10	$1.10	$1.10	$1.10	$1.10	$1.10	$1.10	$1.10	$1.10	$1.10	$1.10	**$1.10**
Copper Price	US$/lb	$2.00					$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	**$2.00**
Indium Price	US$/kg	$500.00					$500.00	$500.00	$500.00	$500.00	$500.00	$500.00	$500.00	$500.00	$500.00	$500.00	$500.00	**$500.00**
Gold Price	US$/oz	$850.00					$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	**$850.00**
Silver Price	US$/oz	$12.00					$12.00	$12.00	$12.00	$12.00	$12.00	$12.00	$12.00	$12.00	$12.00	$12.00	$12.00	**$12.00**
Material Processed																		
Mill Feed	tonnes						84,000	1,210,800	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	759,300	**12,134,101**
Zinc	%						5.70%	5.70%	5.60%	6.40%	5.70%	5.10%	5.30%	5.30%	4.30%	4.30%	3.40%	**5.18%**
Copper	%						0.40%	0.30%	0.40%	0.40%	0.30%	0.30%	0.40%	0.50%	0.50%	0.30%	0.40%	**0.38%**
Indium	g/t						37.00	29.70	22.20	25.40	41.30	49.10	38.60	33.00	35.10	38.40	37.90	**34.99**
Metal Contained in Ore																		
Zinc	lbs						10,555,625	152,151,792	155,556,576	177,778,944	158,334,372	141,667,596	147,223,188	147,223,188	119,445,228	119,445,228	56,914,395	**1,386,296,131**
Copper	lbs						740,746	8,007,989	11,111,184	11,111,184	8,333,388	8,333,388	11,111,184	13,888,980	13,888,980	8,333,388	6,695,811	**101,556,222**
Indium	kg						3,108	35,961	27,972	32,004	52,038	61,866	48,636	41,580	44,226	48,384	28,777	**424,552**
Recovery																		
Zinc Recovery to Zinc Concentrate	%	Formula					84.3%	84.3%	84.1%	85.4%	84.3%	83.3%	83.6%	83.6%	82.1%	82.1%	80.7%	**83.64%**
Copper Recovery to Copper Concentrate	%	80.0%					80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	**80.00%**
Indium Recovery to Zinc Concentrate	%	Formula					57.1%	57.1%	57.3%	55.7%	57.1%	58.3%	57.9%	57.9%	59.8%	59.8%	61.5%	**58.23%**
Concentrate																		
Zinc Concentrate																		
Contained Zinc	lbs						8,894,697	128,210,707	130,838,636	151,734,329	133,420,459	118,058,691	123,144,836	123,144,836	98,058,560	98,058,560	45,929,916	**1,159,494,226**
Contained Indium	kg						1,775	20,533	16,026	17,842	29,713	36,041	28,146	24,063	26,448	28,934	17,709	**247,229**
Concentrate Zinc Grade	%	52.3%					52.3%	52.3%	52.3%	52.3%	52.3%	52.3%	52.3%	52.3%	52.3%	52.3%	52.3%	**52.30%**
Concentrate Indium Grade	g/t						230	185	141	136	257	352	264	225	311	340	445	**246**
Zinc Concentrate	dmt						7,714	111,197	113,476	131,599	115,715	102,392	106,803	106,803	85,046	85,046	39,835	**1,005,627**
Zinc Concentrate	wmt	8%					8,385	120,866	123,344	143,042	125,778	111,296	116,091	116,091	92,441	92,441	43,299	**1,093,073**
Copper Concentrate																		
Contained Copper	lbs						592,596	6,406,391	8,888,947	8,888,947	6,666,710	6,666,710	8,888,947	11,111,184	11,111,184	6,666,710	5,356,649	**81,244,977**
Contained Gold	ounces						567	6,131	8,507	8,507	6,381	6,381	8,507	10,634	10,634	6,381	5,127	**77,758**
Contained Silver	ounces						81,024	875,924	1,215,356	1,215,356	911,517	911,517	1,215,356	1,519,195	1,519,195	911,517	732,397	**11,108,353**
Contained Indium	kg						277	2,997	4,158	4,158	3,119	3,119	4,158	5,198	5,198	3,119	2,506	**38,004**
Concentrate Copper Grade	%	32.0%					32%	32%	32%	32%	32%	32%	32%	32%	32%	32%	32%	**32.00%**
Concentrate Gold Grade	g/t	21.0					21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	**21.0**
Concentrate Silver Grade	g/t	3000.0					3,000.0	3,000.0	3,000.0	3,000.0	3,000.0	3,000.0	3,000.0	3,000.0	3,000.0	3,000.0	3,000.0	**3000.0**
Concentrate Indium Grade	g/t	330					330.0	330.0	330.0	330.0	330.0	330.0	330.0	330.0	330.0	330.0	330.0	**10.3**
Copper Concentrate	dmt						840	9,081	12,600	12,600	9,450	9,450	12,600	15,750	15,750	9,450	7,593	**115,164**
Copper Concentrate	wmt	8%					913	9,871	13,696	13,696	10,272	10,272	13,696	17,120	17,120	10,272	8,253	**125,178**
Unit Deductions																		
Zinc																		
% Contained Zinc	lbs	15.0%					1,334,205	19,231,606	19,625,795	22,760,149	20,013,069	17,708,804	18,471,725	18,471,725	14,708,784	14,708,784	6,889,487	**173,924,134**
Unit Deduction	units	8.0%					1,360,566	19,611,580	20,013,558	23,209,840	20,408,483	18,058,691	18,836,686	18,836,686	14,999,397	14,999,397	7,025,609	**177,360,494**
Zinc Deduction Taken	lbs						1,360,566	19,611,580	20,013,558	23,209,840	20,408,483	18,058,691	18,836,686	18,836,686	14,999,397	14,999,397	7,025,609	**177,360,494**
Copper																		
% Contained Copper	lbs	3.4%					19,852	214,614	297,780	297,780	223,335	223,335	297,780	372,225	372,225	223,335	179,448	**2,721,707**
Unit Deduction	units	1.0%					18,519	200,200	277,780	277,780	208,335	208,335	277,780	347,225	347,225	208,335	167,395	**2,538,906**
Copper Deduction Taken	lbs						19,852	214,614	297,780	297,780	223,335	223,335	297,780	372,225	372,225	223,335	179,448	**2,721,707**
Indium																		
5% of Contained Indium	kg	5.0%					89	1,027	801	892	1,486	1,802	1,407	1,203	1,322	1,447	885	**12,361**
10 g/t Unit Deduction	kg	10.0					77	1,112	1,135	1,316	1,157	1,024	1,068	1,068	850	850	398	**10,056**
Indium Deduction Taken	kg						89	1,112	1,135	1,316	1,486	1,802	1,407	1,203	1,322	1,447	885	**13,204**
Gold Deduction	oz/tonne	0.04					34	363	504	504	378	378	504	630	630	378	304	**4,607**
Silver Deduction	oz/tonne	1.0					840	9,081	12,600	12,600	9,450	9,450	12,600	15,750	15,750	9,450	7,593	**115,164**
Payable Metal																		
Zinc	lbs						7,534,132	108,599,127	110,825,078	128,524,489	113,011,975	100,000,000	104,308,149	104,308,149	83,059,163	83,059,163	38,904,308	**982,133,733**
Copper	lbs						572,744	6,191,777	8,591,167	8,591,167	6,443,376	6,443,376	8,591,167	10,738,959	10,738,959	6,443,376	5,177,201	**78,523,271**
Indium	kg						1,686	19,421	14,891	16,526	28,228	34,239	26,739	22,860	25,125	27,487	16,824	**234,025**
Gold	ounces						566	6,120	8,491	8,491	6,368	6,368	8,491	10,614	10,614	6,368	5,117	**77,610**
Silver	ounces						80,997	875,632	1,214,951	1,214,951	911,213	911,213	1,214,951	1,518,688	1,518,688	911,213	732,152	**11,104,650**
Penalties																		
Zinc Concentrate Penalty	$/dmt	$0.00					$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Copper Concentrate Penalty	$/dmt	$0.00					$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Total Penalties	US$						$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Treatment Charges																		
Zinc																		
Base	US$/dmt	$250.00					$250	$250	$250	$250	$250	$250	$250	$250	$250	$250	$250	
Base Zinc Price	US$/tonne	$2,000					$2,000	$2,000	$2,000	$2,000	$2,000	$2,000	$2,000	$2,000	$2,000	$2,000	$2,000	
Escalator	US$/dmt	$0.09					$0.090	$0.090	$0.090	$0.090	$0.090	$0.090	$0.090	$0.090	$0.090	$0.090	$0.090	
Zinc Treatment Charge	US$						($2,223,705)	($32,053,116)	($32,710,107)	($37,934,102)	($33,355,572)	($29,515,077)	($30,786,631)	($30,786,631)	($24,514,976)	($24,514,976)	($11,482,636)	**($289,877,529)**
Copper																		
Base	US$/dmt	$50.00					$50	$50	$50	$50	$50	$50	$50	$50	$50	$50	$50	
Refining charge	US$/lb	$0.050					$0.050	$0.050	$0.050	$0.050	$0.050	$0.050	$0.050	$0.050	$0.050	$0.050	$0.050	
Copper Treatment Charge	US$						($70,637)	($763,639)	($1,059,558)	($1,059,558)	($794,669)	($794,669)	($1,059,558)	($1,324,448)	($1,324,448)	($794,669)	($638,510)	**($9,684,364)**
Total Treatment Charges							($2,294,342)	($32,816,755)	($33,769,666)	($38,993,660)	($34,150,240)	($30,309,746)	($31,846,189)	($32,111,079)	($25,839,424)	($25,309,645)	($12,121,147)	**($299,561,892)**
Refining Charges																		
Indium - unknown	$/kg	$25.00					$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	**$25.00**
Total Refining Charges							($42,148)	($485,532)	($372,274)	($413,140)	($705,688)	($855,980)	($668,471)	($571,491)	($628,130)	($687,185)	($420,591)	**($5,850,629)**
Gross Revenue after Smelter Charges																		
Zinc Revenue	US$						$6,063,840	$87,405,923	$89,197,479	$103,442,836	$90,957,601	$80,484,923	$83,952,334	$83,952,334	$66,850,103	$66,850,103	$31,312,102	**$790,469,577**
Copper Revenue	US$						$1,074,852	$11,619,915	$16,122,777	$16,122,777	$12,092,082	$12,092,082	$16,122,777	$20,153,471	$20,153,471	$12,092,082	$9,715,892	**$147,362,178**
Indium Revenue	US$						$800,805	$9,225,101	$7,073,205	$7,849,660	$13,408,071	$16,263,626	$12,700,945	$10,858,321	$11,934,475	$13,056,520	$7,991,222	**$111,161,951**
Gold Revenue	US$						$481,173	$5,201,822	$7,217,593	$7,217,593	$5,413,195	$5,413,195	$7,217,593	$9,021,991	$9,021,991	$5,413,195	$4,349,459	**$65,968,799**
Silver Revenue	US$						$971,961	$10,507,589	$14,579,409	$14,579,409	$10,934,557	$10,934,557	$14,579,409	$18,224,262	$18,224,262	$10,934,557	$8,785,830	**$133,255,802**
Total Gross Revenue after Smelter Charges	US$						**$9,392,630**	**$123,960,351**	**$134,190,462**	**$149,212,275**	**$132,805,507**	**$125,188,383**	**$134,573,058**	**$142,210,378**	**$126,184,302**	**$108,346,457**	**$62,154,505**	**$1,248,218,307**
Product Transportation Costs																		
Zinc Concentrate Transportation - US$/wmt basis	US$	$75.00					($628,888)	($9,064,967)	($9,250,771)	($10,728,174)	($9,433,316)	($8,347,182)	($8,706,792)	($8,706,792)	($6,933,100)	($6,933,100)	($3,247,414)	**($81,980,495)**
Copper Concentrate Transportation - US$/wmt basis	US$	$22.50					($20,543)	($222,090)	($308,152)	($308,152)	($231,114)	($231,114)	($308,152)	($385,190)	($385,190)	($231,114)	($185,698)	**($2,816,511)**
Total Transportation Costs	US$						**($649,431)**	**($9,287,057)**	**($9,558,924)**	**($11,036,326)**	**($9,664,430)**	**($8,578,297)**	**($9,014,944)**	**($9,091,982)**	**($7,318,290)**	**($7,164,214)**	**($3,433,112)**	**($84,797,006)**
Royalties																		
Royalty	US$	1.50%					($140,889)	($1,859,405)	($2,012,857)	($2,238,184)	($1,992,083)	($1,877,826)	($2,018,596)	($2,133,156)	($1,892,765)	($1,625,197)	($932,318)	**($18,723,275)**
One time Cash Payment	US$						($1,000,000)											**($1,000,000)**
Total Royalties and Payments	US$						**($1,140,889)**	**($1,859,405)**	**($2,012,857)**	**($2,238,184)**	**($1,992,083)**	**($1,877,826)**	**($2,018,596)**	**($2,133,156)**	**($1,892,765)**	**($1,625,197)**	**($932,318)**	**($19,723,275)**
Net Smelter Return																		
Net Smelter Return							**$7,602,310**	**$112,813,888**	**$122,618,681**	**$135,937,765**	**$121,148,994**	**$114,732,260**	**$123,539,518**	**$130,985,241**	**$116,973,247**	**$99,557,046**	**$57,789,076**	**$1,143,698,027**
Net Smelter Return	*% in-situ value*						*45.46%*	*50.40%*	*51.42%*	*51.31%*	*50.43%*	*50.59%*	*51.55%*	*52.51%*	*53.11%*	*50.89%*	*52.94%*	***51.40%***
Net Smelter Return	*US$/tonne*						*$90.50*	*$93.17*	*$97.32*	*$107.89*	*$96.15*	*$91.06*	*$98.05*	*$103.96*	*$92.84*	*$79.01*	*$76.11*	***$94.25***
Operating Cost Components																		
Mining Costs	US$/t ore						$109.52	$36.34	$35.32	$29.60	$35.32	$33.41	$32.30	$28.65	$28.89	$26.51	$32.27	**$32.36**
Processing Costs (US$/tonne of ore)	US$/t ore	$21.66					$21.66	$21.66	$21.66	$21.66	$21.66	$21.66	$21.66	$21.66	$21.66	$21.66	$21.66	**$21.66**
Tailings Water Management (US$/tonne of ore)	US$/t ore	$0.25					$0.25	$0.25	$0.25	$0.25	$0.25	$0.25	$0.25	$0.25	$0.25	$0.25	$0.25	**$0.25**
G&A Costs (US$/tonne of ore)	US$/t ore	$10.00					$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	**$10.00**
Total Operating Cost per Tonne	US$/t ore	$64.28					$141.44	$68.25	$67.23	$61.52	$67.23	$65.33	$64.21	$60.56	$60.80	$58.42	$64.18	**$64.28**
Operating Costs																		
Mining Costs	US$						($9,200,000)	($44,000,000)	($44,500,000)	($37,300,000)	($44,500,000)	($42,100,000)	($40,700,000)	($36,100,000)	($36,400,000)	($33,400,000)	($24,500,000)	**($392,700,000)**
Processing Costs	US$						($1,819,440)	($26,225,928)	($27,291,600)	($27,291,600)	($27,291,600)	($27,291,600)	($27,291,600)	($27,291,600)	($27,291,600)	($27,291,600)	($16,446,438)	**($262,824,606)**
Tailings Water Management	US$						($21,280)	($306,739)	($319,203)	($319,203)	($319,203)	($319,203)	($319,203)	($319,203)	($319,203)	($319,203)	($192,358)	**($3,074,001)**
G&A Costs	US$						($840,000)	($12,108,000)	($12,600,000)	($12,600,000)	($12,600,000)	($12,600,000)	($12,600,000)	($12,600,000)	($12,600,000)	($12,600,000)	($7,593,000)	**($121,341,000)**
Total Operating Costs	US$						**($11,880,720)**	**($82,640,667)**	**($84,710,803)**	**($77,510,803)**	**($84,710,803)**	**($82,310,803)**	**($80,910,803)**	**($76,310,803)**	**($76,610,803)**	**($73,610,803)**	**($48,731,796)**	**($779,939,607)**
Expensed Development Costs																		
Expensed Development Costs	US$		($7,950,000)	($1,950,000)	($5,190,000)	($6,330,000)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**($21,420,000)**
EBITDA																		
EBITDA	US$		**($7,950,000)**	**($1,950,000)**	**($5,190,000)**	**($6,330,000)**	**($4,278,411)**	**$30,173,222**	**$37,907,878**	**$58,426,962**	**$36,438,191**	**$32,421,457**	**$42,628,715**	**$54,674,438**	**$40,362,444**	**$25,946,243**	**$9,057,280**	**$342,338,420**
Cumulative EBITDA	US$		**($7,950,000)**	**($9,900,000)**	**($15,090,000)**	**($21,420,000)**	**($25,698,411)**	**$4,474,811**	**$42,382,689**	**$100,809,651**	**$137,247,842**	**$169,669,300**	**$212,298,015**	**$266,972,453**	**$307,334,897**	**$333,281,140**	**$342,338,420**	
Deductions																		
Amortization	US$		($1,590,000)	($390,000)	($1,038,000)	($1,266,000)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**($4,284,000)**
Depreciation	US$		($3,407,143)	($4,242,857)	($16,852,033)	($30,942,936)	($41,257,222)	($42,151,400)	($42,468,579)	($41,022,845)	($41,032,881)	($30,599,400)	($17,325,675)	($8,544,227)	($7,667,227)	($7,986,217)	$0	**($335,500,640)**
Depletion	US$		$0	$0	$0	$0	$0	$0	$0	($7,995,781)	$0	($298,444)	($12,002,335)	($19,647,786)	($15,731,140)	($8,433,112)	($4,286,887)	**($68,395,485)**
Severance and Property Tax																		
% of "Gross Proceeds"	US$	30%	$0	$0	$0	$0	$2,817,789	$37,188,105	$40,257,139	$44,763,682	$39,841,652	$37,556,515	$40,371,917	$42,663,114	$37,855,290	$32,503,937	$18,646,352	**$374,465,492**
Annual Exemption	US$	($50,000)	$0	$0	$0	$0	($50,000)	($50,000)	($50,000)	($50,000)	($50,000)	($50,000)	($50,000)	($50,000)	($50,000)	($50,000)	($50,000)	**($550,000)**
Taxable Amount	US$		$0	$0	$0	$0	$2,767,789	$37,138,105	$40,207,139	$44,713,682	$39,791,652	$37,506,515	$40,321,917	$42,613,114	$37,805,290	$32,453,937	$18,596,352	**$373,915,492**
Severance Tax	US$	2.60%	$0	$0	$0	$0	($71,963)	($965,591)	($1,045,386)	($1,162,556)	($1,034,583)	($975,169)	($1,048,370)	($1,107,941)	($982,938)	($843,802)	($483,505)	**($9,721,803)**
Property Tax (estimate; actual 1.0484% of NPV)	US$	1.05%	($50,000)	($50,000)	($50,000)	($50,000)	($250,000)	($250,000)	($250,000)	($250,000)	($250,000)	($250,000)	($250,000)	($250,000)	($250,000)	($250,000)		**($2,700,000)**
Net Income Before Income Taxes	US$		**($12,997,143)**	**($6,632,857)**	**($23,130,033)**	**($38,588,936)**	**($45,857,595)**	**($13,193,769)**	**($5,856,086)**	**$7,995,781**	**($5,879,272)**	**$298,444**	**$12,002,335**	**$25,124,484**	**$15,731,140**	**$8,433,112**	**$4,286,887**	**($78,263,508)**
Income Taxes																		
Federal Tax	US$		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Utah State Tax	US$		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Total Income Taxes	US$		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Net Income after Taxes	US$		**($12,997,143)**	**($6,632,857)**	**($23,130,033)**	**($38,588,936)**	**($45,857,595)**	**($13,193,769)**	**($5,856,086)**	**$7,995,781**	**($5,879,272)**	**$298,444**	**$12,002,335**	**$25,124,484**	**$15,731,140**	**$8,433,112**	**$4,286,887**	**($78,263,508)**
Add Back Deductions																		
Amortization	US$		$1,590,000	$390,000	$1,038,000	$1,266,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$4,284,000**
Depreciation	US$		$3,407,143	$4,242,857	$16,852,033	$30,942,936	$41,257,222	$42,151,400	$42,468,579	$41,022,845	$41,032,881	$30,599,400	$17,325,675	$8,544,227	$7,667,227	$7,986,217	$0	**$327,850,640**
Depletion	US$		$0	$0	$0	$0	$0	$0	$0	$7,995,781	$0	$298,444	$12,002,335	$19,647,786	$15,731,140	$8,433,112	$4,286,887	**$68,395,485**
Capital Costs																		
Initial and Sustaining Capital Costs	US$		($23,850,000)	($5,850,000)	($88,264,230)	($98,636,321)	($52,200,000)	($6,259,250)	($2,220,250)	($13,729,863)	($5,920,250)	($15,229,863)	($5,720,250)	($10,729,863)	($120,250)	($20,250)	($6,750,000)	**($305,800,640)**
Working Capital	US$						($20,000,000)										$20,000,000	**$0**
Total Unexpensed Capital Costs	US$		($23,850,000)	($5,850,000)	($88,264,230)	($98,636,321)	($72,200,000)	($6,259,250)	($2,220,250)	($13,729,863)	($5,920,250)	($15,229,863)	($5,720,250)	($10,729,863)	($120,250)	($20,250)	$13,250,000	**($305,801,529)**
Cash Flow after Taxes	US$		**($31,850,000)**	**($7,850,000)**	**($93,504,230)**	**($105,016,321)**	**($76,800,373)**	**$22,698,381**	**$34,392,243**	**$43,284,543**	**$29,233,358**	**$15,966,425**	**$35,610,095**	**$42,586,634**	**$39,009,256**	**$24,832,191**	**$21,823,775**	**($5,584,023)**
Cumulative Cash Flow after Taxes	US$		**($31,850,000)**	**($39,700,000)**	**($133,204,230)**	**($238,220,551)**	**($315,020,924)**	**($292,322,543)**	**($257,930,300)**	**($214,645,757)**	**($185,412,399)**	**($169,445,974)**	**($133,835,878)**	**($91,249,244)**	**($52,239,988)**	**($27,407,797)**	**($5,584,023)**	
After Tax																		
NPV @ 2%	US$	2.0%	**($2,790,986)**															
NPV @ 4%	US$	4.0%	**($31,770,446)**															
NPV @ 5%	US$	5.0%	**($43,836,366)**															
NPV @ 6%	US$	6.0%	**($54,526,375)**															
NPV @ 8%	US$	8.0%	**($72,372,323)**															
NPV @ 10%	US$	10.0%	**($86,326,962)**															
NPV @ 12%	US$	12.0%	**($97,184,815)**															
NPV @ 15%	US$	15.0%	**($108,991,152)**															
IRR			1.8%															
Initial Development Capital (including working capital)			$186,900,551															
Total Development Capital			$305,800,640															
Payback (from Start of Construction)			14.0 Years															
Payback (from End of Construction)			12.0 Years															
Before Tax																		
EBITDA plus Unexpensed Capital Costs			($31,800,000)	($7,800,000)	($93,454,230)	($104,966,321)	($76,478,411)	$23,913,972	$35,687,628	$44,697,099	$30,517,941	$17,191,595	$36,908,465	$43,944,575	$40,242,194	$25,925,993	$22,307,280	**$6,837,780**
Cumulative EBITDA plus UnExpensed Capital Costs			**($31,800,000)**	**($39,600,000)**	**($133,054,230)**	**($238,020,551)**	**($314,498,961)**	**($290,584,990)**	**($254,897,362)**	**($210,200,263)**	**($179,682,321)**	**($162,490,727)**	**($125,582,262)**	**($81,637,687)**	**($41,395,493)**	**($15,469,500)**	**$6,837,780**	
NPV @ 2%	US$	2.0%	**$7,752,289**															
NPV @ 4%	US$	4.0%	**($22,694,300)**															
NPV @ 5%	US$	5.0%	**($35,397,036)**															
NPV @ 6%	US$	6.0%	**($46,668,188)**															
NPV @ 8%	US$	8.0%	**($65,531,419)**															
NPV @ 10%	US$	10.0%	**($80,340,706)**															
NPV @ 12%	US$	12.0%	**($91,920,604)**															
NPV @ 15%	US$	15.0%	**($104,612,224)**															
IRR			2.5%															
Initial Development Capital (including working capital)			$186,900,551															
Total Development Capital			$305,800,640															
Payback (from Start of Construction)			12.7 Years															
Payback (from End of Construction)			10.7 Years															
Cash Cost (US$/lb payable Zn; net of credits)							**$1.24**	**$0.53**	**$0.46**	**$0.35**	**$0.48**	**$0.48**	**$0.40**	**$0.28**	**$0.32**	**$0.49**	**$0.57**	**$0.43**
Total Cost (US$/lb payable Zn; net of credits)							**$6.76**	**$0.93**	**$0.86**	**$0.68**	**$0.86**	**$0.80**	**$0.57**	**$0.38**	**$0.43**	**$0.60**	**$0.58**	**$0.81**